Horizonte Minerals Plc

NI43-101 Technical Report – Vermelho Project, Pará State, Brazil

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SNOWDEN

www.snowdengroup.com

OFFICE LOCATIONS

Perth

Level 6, 130 Stirling Street
Perth WA 6000
AUSTRALIA

Tel: +61 8 9213 9213
ABN: 99 085 319 562
perth@snowdengroup.com

Brisbane

22 Cordelia Street
South Brisbane QLD 4101
AUSTRALIA

Tel: +61 7 3026 6666
ABN: 99 085 319 562
brisbane@snowdengroup.com

Johannesburg

Technology House, Greenacres
Office Park, Cnr. Victory and
Rustenburg Roads, Victory Park
Johannesburg 2195
SOUTH AFRICA

PO Box 521278, Saxonwold 2132
SOUTH AFRICA

Tel: +27 11 782 2379
Fax: +27 11 782 2396
Reg. No. 1998/023556/07
johannesburg@snowdengroup.com

Website

www.snowdengroup.com

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Prepared by: Mr. Anthony Finch
B Eng (Min), B Econ, MAusIMM (CP Min), P Eng (APEGBC)
Executive Consultant,
Snowden Mining Industry Consultants

Mr. Andrew F. Ross
M Sc, FAusIMM
Executive Consultant,
Snowden Mining Industry Consultants

Mr. Simon Walsh
B Sc (Extr. Met.& Chem), MBA, MAusIMM (CP), GAICD
Principal Metallurgist,
Simulus Pty Ltd

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1 SUMMARY

1.1 Introduction

This report is a National Instrument 43-101 (NI 43-101) Technical Report on the Prefeasibility Study (PFS) of the Horizonte Minerals Plc (HZM or “the Company”) wholly owned Vermelho Nickel-Cobalt Project (Vermelho or “the Project”). The Project is located in the north-western Brazilian state of Pará in the Carajás Municipality, approximately 760 kilometres (km) south of the state capital Belém. The term “Prefeasibility Study” has the meaning ascribed by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended. Under CIM guidelines, the Project is considered to be a “development property” – a property that is being prepared for mineral production and for which economic viability has been demonstrated by a PFS.

The Project was first developed by Vale (formerly CVRD) with the objective of becoming its principal nickel-cobalt operation. Extensive work was undertaken on the Project at Scoping (Preliminary Economic Assessment or PEA), PFS and Feasibility Study (FS) stages. This included drilling and pitting programs totalling 152,000 m, batch and full-scale pilot testwork and detailed engineering studies. The Project was subsequently taken through a Feasibility program with Vale reporting a positive development decision in 2005. The Project was designed around the construction of a High-Pressure Acid Leach (HPAL) plant to process the nickel/cobalt laterite ores. The FS included a five-year metallurgical testwork and pilot plant program which demonstrated 96% average leach extraction rates of nickel and cobalt. In addition, London Metal Exchange (LME) grade nickel (metal) cathode was produced. The FS proposed production capacity was 46,000 tonnes per annum (t/a) of metallic nickel, and 2,500 t/a of metallic cobalt, with an expected commercial life of 40 years. The Project was subsequently placed on hold after delivery of the FS due to Vale liquidity issues.

The Project comprises a planned open pit nickel laterite mining operation that mines a 141.3 million tonnes (Mt) Probable Mineral Reserve of a 145.7 Mt Measured and Indicated Mineral Resource contained within the mining licence (at a cut-off of 0.7% Ni) to produce 924 thousand tonnes (kt) of nickel contained in nickel sulphate, 36 kt of cobalt contained in cobalt sulphate and a saleable by-product, kieserite (a form of fertiliser) of which 4.48 Mt are produced. The Project has a 38-year life. The hydro-metallurgical process comprises a beneficiation plant where ore is upgraded prior to being fed to a HPAL plant which produces the sulphates. The plant will be constructed in two phases, with an initial capacity of 1 Mt per annum (Mt/a) autoclave feed (Stage 1), then after three years of production, a second process train will be constructed effectively doubling the autoclave feed rate to 2 Mt/a (Stage 2 expansion). The Stage 1 plant and project infrastructure will be constructed over a 31-month period. The nickel and cobalt sulphate products will be transported by road to the port of Vila do Conde for sale to overseas customers. The kieserite will be transported to consumers within Pará state.

The process plant, mining, infrastructure and utilities engineering has been developed to support capital expenditure (capex) and operating expenditure (opex) estimates to the Association for the Advancement of Cost Engineering (AACE) class 4 standard. This means that capex and opex estimates have a combined accuracy of between -25% and +20% with a confidence level of 50%. The capex and opex are as of Q2 2019.

This report has been prepared in accordance with the terminology, definitions and guidelines in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

Simulus Pty Ltd (Simulus), Snowden Mining Industry Consultants Pty Ltd (Snowden), WALM Engenharia e Tecnologia Ambiental Ltda (WALM), and C. Steinweg Handelsveem (Latin America) S.A. (Steinweg) were the key consultants commissioned by HZM to produce the PFS for the Project.

The PFS concludes that the Project is economically and technically viable and should be progressed to Feasibility, and that the Project has sufficient Mineral Reserves to support a 38-year life of mine (LOM). The Project has an NPV_8 of over US$1.7 billion and returns an internal rate of return (IRR) of over 26% on an initial capex of US$652.2 million.

Metric units have been used throughout this report. All currency values are expressed in United States dollars (US$) exclusively. The Project assumed that 1 US$ buys 3.8 Brazilian Reais (BRL).

1.2 Property description and ownership

The Project area is characterised by relatively flat terrain, with two hills forming the V1 and V2 deposits. The hydrographical basin is formed by the Araras and the Verde rivers, both of which are tributaries of the Parauapebas River.

The original vegetation is the equatorial latifoliated forest, with transitions to a tropical forest, dominated by low and medium size plants, and locally with very tall trees. Some decades ago, most of the area has been deforested, and is currently used for agriculture purposes.

The Vermelho licence is located near the town of Canaã dos Carajás within the Carajás Mining District. Originally an Exploration Licence, the Vermelho licence is now described by ANM Process 808.055/1974 and covers 2,000 hectares (ha) including the V1 and V2 nickel-cobalt laterite deposits.

The mineral rights phase is in between the first stage (Exploration) and the second stage (Mining) – i.e. in the period where a mining licence application (or “Requerimento de Lavra”) is submitted to and decided by the “Departmento Nacional de Produção Mineral” (ANM). Currently, the Mineral Rights process is pending a decision by the ANM regarding the revised “Plano de Aproveitamento Econômico da Jazida” (economic development plan or “PAE”) submitted by HZM in March 2019.

Access to the area is by plane from Belo Horizonte (Minas Gerais) to Carajás, then 95 km by paved State road to Canaã dos Carajás. The licence area is traversible in a 4x4 vehicle along a series of farm tracks.

1.3 Geology and mineralisation

The Vermelho nickel deposits consist of two hills named V1 and V2 (after Vermelho 1 and Vermelho 2), aligned on a northeast-southwest trend, overlying ultramafic bodies. A third ultramafic body, named V3, also located in the same trend, lies on flat terrain, southwest of V2. The ultramafic bodies have had an extensive history of tropical weathering, which has produced a thick lateritic profile of nickel-enriched limonite and saprolite at V1 and V2.

The V1 and V2 deposits form flat lying topographical highs, where the V1 hill reaches heights of around 500 m, standing out from the adjacent terrain about 175m whereas the V2 hill has a maximum height of 450 m. The flattened terrain adjacent to the two bodies comprises the gneissic ground that hosts the layered intrusion of Vermelho. The V2 hill is located approximately 2 km southwest of V1. The V1 hill has a deformed convex-concave shape (convex to northeast), and extends for approximately 2.4 km east-west, ranging from 700 m to 1.6 km north-south. The V2 hill has an east-west elongation, and extends for approximately 1.9 km east-west, ranging from 600 m to 900 m north-south. The V1 deposit has an average thickness of 53 m and a maximum thickness of 146 m, whereas the V2 deposit has an average thickness of 56 m and a maximum thickness of 115 m.

The ultramafic bodies are erosional remnants of the upper sheet of a three-layer intrusion, represented from bottom to top, by a mafic zone (consisting of gabbros, gabbro-norites and leuconorites), a pyroxenitic zone (orthopyroxenites and chromitites, of c. 50 m thickness) and a peridotitic zone (serpentinised dunites and harzburgites, of c. 150 m thickness). The northeast-southwest oriented, 10 km long and 2 km wide, layered intrusion has intruded a package of gneisses and migmatites belonging to the Xingu Complex. Late sub-vertical diabase dykes intersect the three layers in different directions. Various chromitite levels have been identified at the southern sides of both V1 and V2 within the pyroxenitic zone.

The sequence has undergone hydration as the magmatic chamber aged. Dunites and peridotites are strongly serpentinised, and pyroxenites are amphibolitised. The silicification which supports and preserves the topographic relief is the starting point of the weathering processes.

The structural features at the Project were fundamental to the evolution of the weathering and consequently to the mineralisation. The horizontal to sub-horizontal foliation planes in association with the vertical joints and fractures worked as conduits for water migration during weathering and assisted the leaching processes and the precipitation of silica and other elements responsible for the formation of the boxwork texture presented in the silicified saprolite ore.

The weathering evolution of the Vermelho Mafic-Ultramafic Complex resulted in a truncation of lateritic evolution and the resumption of the process after climate change. The formation of lateritic soil was followed by significant silicification, generating a thick layer of chalcedony, erosion of the primary lateritic package and generation of a new lateritic profile under the silicified level. Thus, the profile presents interdigitations of lateritic units with significant thickness variations. These weathering products are described below (from top to bottom):

- The soil (coded COB in drill logs) is the overburden, formed of fine ferruginous soil, organic residues, and ferruginous concretions, with relative enrichment of Fe_2O_3 and Al_2O_3. Usually under the ferruginous carapace (ferricrete) is 2 m or 3 m of ferruginous limonite (SAPFE), without ferruginous concretions.
- The silica horizon, coded SILICA, formed of thick, very resistant massive silica crusts, with SiO_2 content usually above 65%, sometimes reaching 90%. This also represents intense leaching, but not as intense as that of soil or ferruginous limonite (30 m average thickness).
- Underneath the SILICA is the Silicified limonite layer (SAPSIL), with varying proportions between oxides and hydroxides of iron and silica with SiO_2 content ranging from 35% to 65%, and boxwork structures are usually packed with Fe oxides and hydroxides (20 m average thickness).
- The next layer, ferruginous limonite (SAPFE) is composed of finely grained and porous Fe oxides and hydroxides (goethite, limonite, hematite), and forming discontinuous and irregular lenses within the siliceous limonite or the massive silica.
- The saprolite (SAP) layer composes a constant level overlapping weathering/fresh rock surface. In general, its top is strongly silicified (2–3 m). The contact between limonite and saprolite can be abrupt or transitional.

Since the work that Vale carried out in the PEA, PFS and FS on the Project, the HZM technical team has elected to update the nomenclature of the weathering units so that they are in alignment with the geological descriptions at HZM's other projects in the region.

Vale carried out mineralogical studies in 2003 and 2004 in order to understand the variations among the mineralised horizons and to develop a proposed beneficiation process. Thus, 50 samples previously selected for sulphuric acid leaching tests were characterised.

The beneficiation process was comparatively more efficient in removing quartz from siliceous limonite. In saprolite, the separation of quartz and Mg silicates appears more complex, since both mineral species are found in the two particle size fractions, causing loss of Ni in the tailings and increase of silica in the leaching feed. This may be related to textural aspects, such as high particle size, quartz inclusion and low porosity, due to the poor weathering of saprolite. In addition, Ni appears more dispersed, as it occurs at lower levels in Mg silicates of this unit. The incipient enrichment of ferruginous limonite in the leaching feed would be due to the mineralogical homogeneity, or the low percentage of mineral species other than Fe oxides/hydroxides themselves.

1.4 Status of exploration, development and operations

The Vermelho area was explored in various stages by Companhia Vale do Rio Doce (CVRD, now Vale) from 1974 to 2004 involving approximately 152,000 m of combined drilling and pitting. The drilling grid density was substantially enhanced in 2002 to 2004, and most of the resources were upgraded to the Measured category as defined in the CIM Definition Standards. Pilot plant metallurgical studies were conducted in Australia focused on the HPAL processing method. A PFS was prepared in 2003, and a FS was completed in August 2004 by GRD-Minproc (2005). This study confirmed the positive economic outcomes obtained in previous studies and showed production capacity of 46,000 t/a of metallic nickel, and 2,500 t/a of metallic cobalt. The Project was given construction approval by the Vale board in 2005. Later that year Vale elected to place the Project on hold after it acquired Canadian nickel producer, Inco.

1.5 Mineral Resource estimates

During February 2018, historical Mineral Resource estimates (MREs) were reviewed and it was concluded that the “FFS_25_2_m” MRE that was audited by Snowden in 2005, is appropriate for adoption as a current MRE for HZM. The review concluded that the reporting conforms to the requirements of the CIM Definition Standards (2014).

Within the mining licence, at a cut-off grade of 0.7% Ni, a total of 140.8 Mt at a grade of 1.05% Ni and 0.05% Co is defined as a Measured Mineral Resource; and a total of 5.0 Mt at a grade of 0.99% Ni and 0.06% Co is defined as an Indicated Mineral Resource. This gives a combined tonnage of 145.7 Mt at a grade of 1.05% Ni and 0.05% Co for Measured and Indicated Mineral Resources. A further 3.1 Mt at a grade of 0.96% Ni and 0.04% Co is defined as an Inferred Mineral Resource at a cut-off grade of 0.7% Ni.

The Mineral Resource is summarised in Table 1-1.

Table 1-1 V1 + V2 – combined classified Mineral Resource report for Vermelho above 0.7% Ni cut-off within the mining licence

Classification	Tonnage (Mt)	Ni (%)	Ni metal (kt)	Co (%)	Co metal (kt)	Fe_2O_3 (%)	MgO_2 (%)	SiO_2 (%)
Measured	140.8	1.05	1,477	0.05	74.6	31.1	11.3	41.0
Indicated	5.0	0.99	49	0.06	2.8	26.3	8.6	49.0
Measured + Indicated	145.7	1.05	1,526	0.05	77.3	30.9	11.2	41.3
Inferred	3.1	0.96	29	0.04	1.4	24.0	15.5	42.2

Notes:

- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Snowden does not consider them to be material.*
- *Mineral Resources are reported inclusive of Mineral Reserves.*
- *The reporting standard adopted for the reporting of the Mineral Resource estimate uses the terminology, definitions and guidelines given in the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI 43-101.*

- *Mineral Resources are reported on 100% basis for all Project areas.*
- *Snowden completed a site inspection of the deposit by Mr Andy Ross (FAusIMM), an appropriate "independent qualified person" as such term is defined in NI 43-101.*
- *kt = thousand tonnes (metric).*

1.6 Mining

1.6.1 Mining method

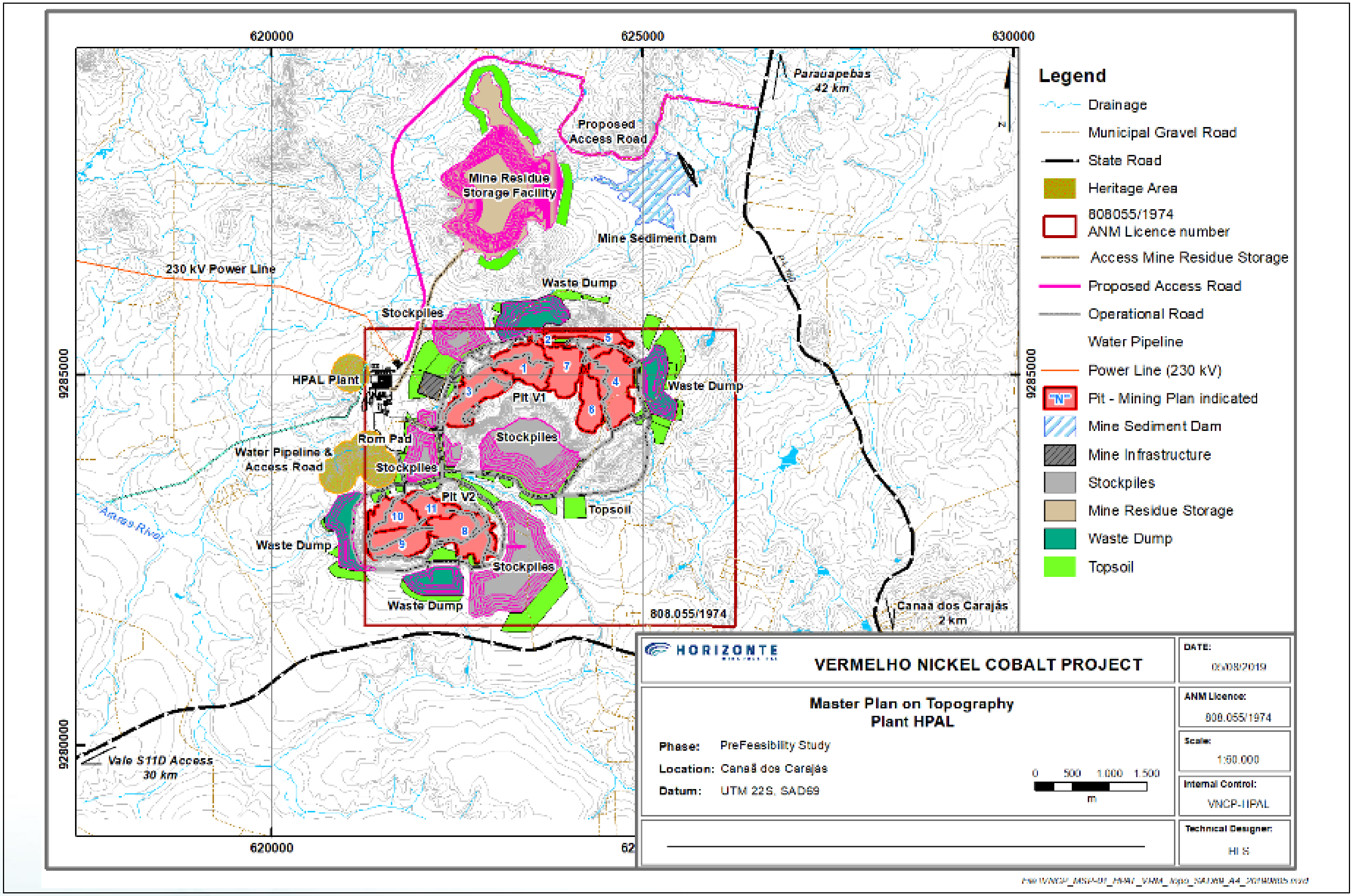
Mining at Vermelho is planned to be undertaken with conventional open pit truck and excavator mining methods. Blasting will be necessary for about half of the deposit. Waste overburden will be stripped on 4 m benches, and ore on 2 m benches for additional selectivity.

Reverse circulation (RC) grade control drilling will be completed at 12.5 m x 12.5 m spacing to define the waste/ore/ore type boundary ahead of mining.

Waste will be stored in dumps adjacent to the pits. Ore will be transported to the run of mine (ROM) stockpile near the processing plant or the low-grade stockpiles for later processing (Figure 1-1).

Due to the high rainfall in the wet season, mining (including stockpile rehandling) will be reduced between October and March (as is standard practice in the region). It was assumed that a fleet of Scania G500 8x4 22 m^3 heavy tipper will be used as part of the fleet and coarse beneficiation rejects will be used as sheeting, to mitigate trafficability issues.

Figure 1-1 Planned site layout



Source: HZM, 2019

1.6.2 Mine design

No further dilution was added to the resource model, which was modelled using blocks of 25 m x 25 m x 2 m, noting the very low strip ratios and large block sizes. A 2% ore loss was assumed. Beneficiation mass pull and upgrade functions were coded into the block model.

The pits were optimised based on the range of assumptions outlined in Section 16. The cut-off grade is dependent on both nickel and cobalt revenue generation and the costs associated with processing (including variable acid consumption and related costs). The pits were optimised using Whittle 4X software to determine a shell to use for design.

The pits were designed in multiple stages to expedite the mining of higher value areas and to ensure access to the top of the deposits. Waste and topsoil dumps were designed to store the necessary capacity. For this Project, the stockpiles are significant at around 84 Mt. Stockpiles separate out limonite and saprolite layers in order to manage magnesium grades.

1.6.3 Mineral Reserve

A summary of the Mineral Reserves is provided in Table 1-2.

Table 1-2 Vermelho Mineral Reserve estimates, as at 31 October 2019

Value	Probable
Ore (Mt)	141.3
Ni (%)	0.91
Co (%)	0.052
Fe (%)	23.1
Mg (%)	3.81
Al (%)	0.79

Notes

- *Cut-off varies by resource model block depending on individual block geochemistry, however, as a guide the cut-off is approximately 0.5% Ni.*
- *Dilution was modelled as part of re-blocking, ore losses applied are 2%.*
- *Snowden completed a site inspection of the deposit by Mr Anthony Finch P.Eng (MAusIMM), an appropriate "independent qualified person" as such term is defined in NI 43-101.*

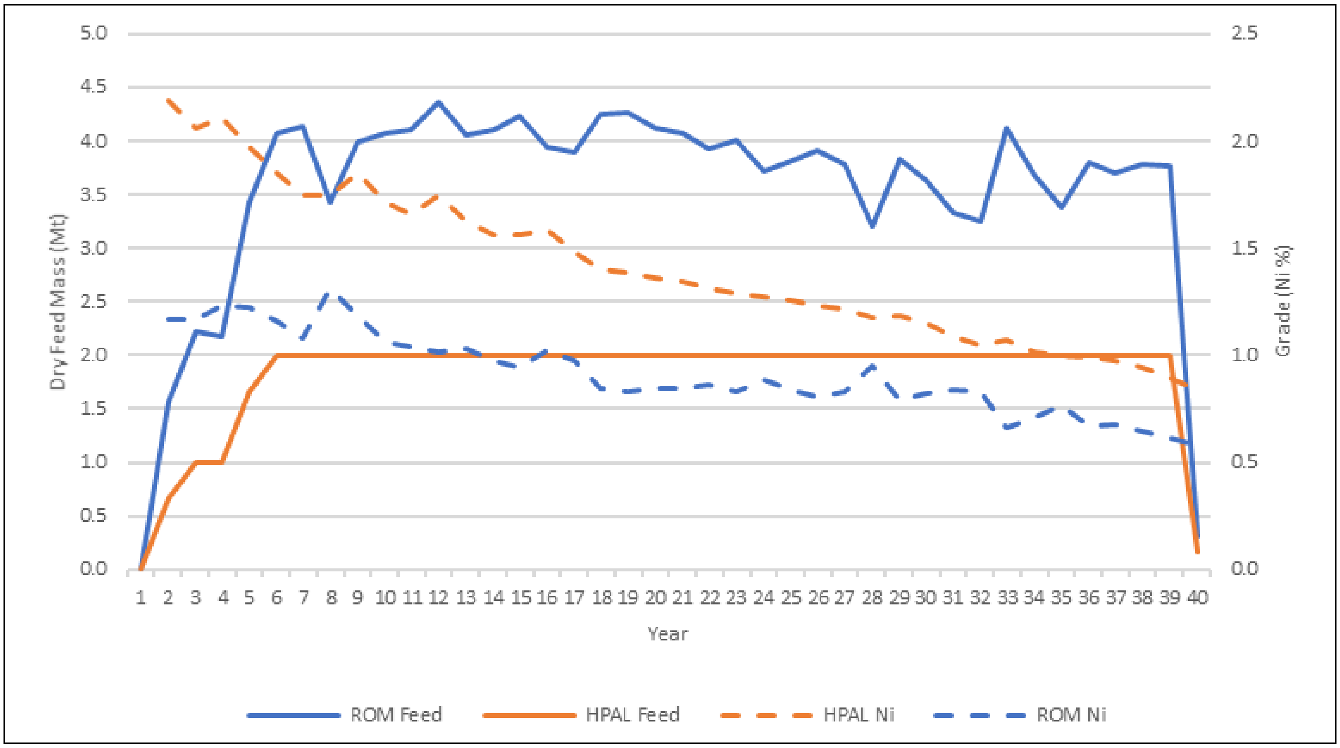
1.6.4 Mine production schedules

The mine production schedule targeted a processing rate of 1 Mt/a HPAL feed for the first three years and doubling in capacity thereafter to 2 Mt/a. To facilitate this, ROM feed of approximately 2.25 Mt/a to 4.5 Mt/a is required as well as an acid production capacity of 350 kt/a to 700 kt/a (Figure 1-2). The scheduling increments were annual for the LOM.

The schedule considered:

- Maximisation of nickel and cobalt revenue
- Capping of magnesium grades to 5%
- Minimisation the number of active deposits
- Smoothing of the overall mining rates.

Figure 1-2 Processing schedule summary



Source: Snowden, 2019

The annual mining rate starts at 8 Mt/a and peaks at 12 Mt/a between production years 5 and 11.

The mine supplies higher grade ore in the early mine life to the HPAL circuit, reaching 2% Ni and 0.1% Co in the first four production years. The HPAL feed grade (after beneficiation) is above 1.5% Ni and 0.08% Co for the majority of the first 17 years of production and reduces over the remaining LOM as feed is sourced from large lower grade stockpiles that were developed in the early years and are depleted in the later years.

1.6.5 Mine requirements

To support the proposed mine plan the following **peak** quantities are required:

- 123 trucks, 15 loaders, 13 excavators, seven dozers, five drills, and eight other items of heavy equipment
- 856 staff in total including operators, management, technical, maintenance and support personnel (includes allowances for annual leave, sick leave, training etc.)
- 870 t/a explosives
- 1,646 kL/a diesel.

1.7 Recovery methods

The process selected for the Project is the production of nickel and cobalt sulphate product via HPAL, mixed sulphide precipitate (MSP), pressure oxidation leaching (POX), cobalt solvent extraction (CoSX) and crystallisation. Prior to the HPAL process barren free silica is removed from the ore via a beneficiation process which involves crushing, scrubbing and washing, separation by screening and by hydro-cyclones.

The flowsheet selection is consistent with the HPAL, MSP, POX and hydrogen reduction option considered during an earlier PFS by the former owners, Vale, but with substitution of crystallisation in place of hydrogen reduction. This option was ranked second behind the mixed hydroxide precipitate (MHP) ammonia re-leach option originally selected by Vale. The justification for the change to the final unit processes being the emergence of the nickel and cobalt sulphate market and the price premium available for these products, as well as the reduced cost of the crystallisation process compared to hydrogen reduction and the improved rejection of impurities via the MSP intermediate compared to the MHP/ammonia re-leach process. The HPAL, MSP, POX and CoSX processes are all considered conventional, low risk, industrially proven, robust processes.

Simulus has completed a PFS-level engineering design including full mass and energy balance using SysCAD process modelling software, process flow diagrams, three-dimensional (3D) plant model and layout, mechanical equipment list, electrical load list, equipment and material offtakes to support capital and operating cost estimates.

The process plant has been designed to process 4.34 Mt/a of ROM ore at 1.07% Ni. Of this total feed, 2.34 Mt/a is rejected as coarse, low grade siliceous waste from the beneficiation plant. The beneficiated 2 Mt/a product at 1.85% Ni is fed to the HPAL processing plant as an upgraded feed (1 Mt/a per process train). A common refining circuit treats the MSP produced from each train via POX, CoSX and crystallisation. The crushing circuit was designed for 75% availability and all downstream process plant was designed for 85% availability.

The Project implementation plan includes installation of one train of crushing and beneficiation, HPAL and associated 1,200 tonnes per day (t/d) acid plant and the common refining section (Stage 1). This is followed by the addition of a second crushing and beneficiation, HPAL and 1,200 t/d acid plant train in year 4 (the Stage 2 expansion).

The proposed process plant has been designed to recover 94.4% and 94.9% of the nickel and cobalt from the HPAL feed with an average acid consumption of 347 kg/t. The final nickel and cobalt sulphate products are of high purity suitable for sale directly into the battery market.

A kieserite (magnesium sulphate monohydrate or $MgSO_4{\bullet}H_2O$) by-product suitable for use in the fertiliser industry will also be produced as part of the zero liquid discharge design philosophy. The main waste solids produced will be the benign low-grade beneficiation rejects and neutralised and washed HPAL residue that will be filtered and dry stacked in the residue storage facility.

1.8 Metallurgical testwork and process development

Extensive metallurgical testwork and process design was undertaken on the Project by the former owner, Vale, at scoping, prefeasibility and feasibility stages, included drilling and pitting programs totalling 152,000 m, variability batch testwork, full-scale pilot testwork and detailed engineering studies. A five-year, exhaustive, metallurgical testwork and pilot plant program demonstrated that a high degree of mined ore upgrade using a simple beneficiation processes was possible. The resultant feed delivered 96% average leaching extraction for nickel and cobalt via HPAL technology.

Additional testwork has been completed by the current Project owner, HZM, during 2018 and 2019. This testwork on selected samples from Vermelho validated the potential to produce high-grade sulphate products using the HPAL process.

The 6,000 plus samples totalling over 160 t used for PFS and Final Feasibility Study (FFS) piloting were large diameter drill core and were representative (geographically, of depth, ore type and by lithology). Additionally, 10% of the samples (1 m from every 10 m) was used for variability testing so piloting and variability were related. A detailed review of sample representivity was completed by Geostats Pty Ltd (Geostats, 2004).

PFS piloting performed at Lakefield Oretest in Australia from November 2002 to March 2003 included four beneficiation runs and three HPAL runs consisting of 14 days through to an MHP intermediate product and seven days through to an MSP intermediate product. Equipment vendor and chemical reagent supplier testwork was completed using piloting samples. Variability testwork (220 samples tested for beneficiation and 50 beneficiated samples tested for HPAL) was conducted in parallel to piloting.

FFS pilot testwork was conducted in April to October 2004 also at SGS Lakefield Oretest with the aim of demonstrating the beneficiation, HPAL, MHP, ammonia re-leach and solvent extraction-electrowinning (SX-EW) process. A total of 6,640 x 1 m interval samples were delivered to SGS Lakefield Oretest for pilot testwork, with approximately 550 inter-twinned samples used for batch variability testwork. Eight beneficiation runs of approximately 12.5 t each and four HPAL/MHP campaigns of 14 days each were completed. In addition to previous PFS variability tests, an additional 368 beneficiation and 50 HPAL batch variability tests were completed during the FFS phase.

FFS piloting nickel extractions were typically greater than 96.5%. It is expected that similar extraction should be achievable in the full-scale industrial plant. Cobalt extraction was typically greater than 95.5%. Free acid levels of 55–60 g/l in the leach discharge were required to achieve the stated nickel and cobalt recoveries.

HZM commissioned additional testing which was completed in 2018 and 2019 at SGS Lakefield in Ontario (Canada), where 157 kg (wet) of ore was treated via, HPAL, MHP, acid re-leach CoSX, batch purification steps and crystallisation to make cobalt sulphate. Twenty 20-litre batch HPAL tests were completed and averaged 98% and 96% Ni and Co extraction respectively at a 350 to 400 kg/t acid dose. The MHP precipitation, acid re-leach, CoSX with Cyanex 272, ion exchange and sulphide precipitation followed by crystallisation produced cobalt sulphate crystals with a purity of 99.91%.

1.9 Environmental, social and permitting

The Project is located 3 km from the town of Canaã dos Carajás, founded in 1994, which forms the southern limit of the Carajás Mining District (CMD) Pará state, north of Brazil. The CMD is a host to a number of tier 1 iron, nickel and copper mines operated by Vale.

Mining and related industries in the CMD play a vital role in the socio-economic fabric of the region, with the municipality presenting considerable per capita income, the second highest of the Pará state.

In 2004, Vale started to operate the Sossego Copper Mine after several infrastructure municipality improvements, and most recently (2017) ramped-up the S11D project, one of the largest standalone iron operations in the world. As a result of the advances of mining in the region, there has been a significant influx of people and investment, which has in turn promoted changes and improvements in the areas of economic growth, cultural diversity and a more developed economy than nearby towns, which is heavily centred around mining related activities.

Key environmental studies for the advancement of project licensing stages were completed by Vale. HZM will utilise the studies and baseline data collected by previous owners to inform and expedite new EIA RIMA studies.

The following mining and environmental permits were granted to Vale by the end of 2016:

- EIA RIMA studies (Environmental Impact Study (EIS) and Environmental Impact Report (EIR)) issued
- Award of Preliminary Licence (LP)
- Environmental Controls Plan issued
- Application for Installation Licence (LI)
- Final Exploration Report approved
- Mine Plan (Plano de Aproveitamento Economico – PAE) approved.

Whilst a new permit pathway is proposed (see Section 20.2.6), the previously awarded permits for Vermelho provide a solid foundation from which to progress the project permitting.

HZM will utilise the Vale studies and baseline data collected to inform and expedite new EIA RIMA studies. As HZM will recommence the licensing for Vermelho, the Company will both update studies and undertake new studies to accurately characterise the current physical environment, biological environment and social settings.

1.10 Infrastructure

The site layout around the HPAL plant includes a large dry-stacked tail and rejects co-disposal facility, incoming powerline and roads, and a water pipeline to the plant from the Araras River to the west, and a sediment control dam (see Figure 1-1).

1.10.1 Mine residue storage facility and mine sediment dam

A disposal facility has been designed to house all the dry tails and beneficiation rejects for the Project (approximately 110 million cubic metres or Mm^3). The facility will be lined with 0.5 m of clay and then a 1.5 mm membrane. The facility includes surface and basement drainage. There will be several lifts for the facility throughout the mine life. Dry tailings and rejects will be trucked from the HPAL plant and placed in the facility in layers wherein it will be compacted continuously. Water drained from the facility will be settled in a purpose-built sediment dam.

1.10.2 Water pipeline

The average water requirement for the Project was estimated at 165 m^3/hr. To support this, the Project proposes the construction of a 4.5 km, 300 mm diameter, steel water pipeline including pumping from the Araras River to the plant. A water catchment study and water balance has been completed to support this solution.

1.10.3 Road works

The Project is well served by State highway PA160 which travels north-south to the east of the Project and connects to the national highway system. In order to access the site, HZM will construct approximately 11 km of road that will intersect with PA160 approximately 8 km north of the town of Canaã dos Carajás.

1.10.4 Energy supply

Power demand for the Project is estimated at approximately 18 megawatts (MW) (40 MW if the co-generation plant included in the study is removed). Total installed power is 135 MW. A study completed by Severino Macedo Consultoria e Engenharia (SM&A) concluded that this would be best served by connecting to the SE-Integradora substation owned by Eletrobrás/Eletronorte. The solution includes construction of a new 230 kV bay at SE-Integradora, a new 15 km 230 kV transmission and a new substation in the Project, with one entrance bay and two or three transformer bays.

1.11 Project implementation and schedule

A preliminary Project Implementation Plan has been developed to provide an overview of how the Project may be implemented. In this study, an engineering, procurement, construction and management (EPCM) contracting strategy has been assumed. The EPCM Contractor will assume responsibility for direct management of all engineering, procurement and construction activities on the Project acting as Owners Agent with all contracts and purchases placed directly by the Owner.

The contractors will be responsible for managing major aspects of the development, while HZM retains overall accountability for the development with specific responsibilities for:

- Project funding
- Overall project governance
- Regulatory approvals
- Third party infrastructure interface management
- Regulatory submissions
- Geological exploration, mapping, resource assessment.

A project implementation schedule has been developed for the design and construction of the Project. The schedule relies on the procuring of the long-lead items soon after commencement of the works and completing the design and detailed drafting to allow the fabrication and procurement of less critical components. The schedule is based on 33 months from approval to proceed until practical completion. Commissioning is scheduled for a further three months.

1.12 Capital expenditure estimate

The estimate is based on the AACE class 4 standard, with an estimated accuracy range between -25% and +20% (Table 21-1) of the final project cost (excluding contingency).

The largest capital item is the HPAL plant. In order to reduce **initial** capital, this is constructed in two phases. The first phase (Stage 1) has a capacity of 1 Mt/a autoclave feed. The Stage 2 expansion is brought online in year 3 of production and effectively doubles the HPAL feed rate to 2 Mt/a.

A summary of the Project capital cost is provided in Table 1-3.

Table 1-3 Capital cost summary

Capital cost component	Stage 1 (initial) (US$ M)	Stage 2 (year 3) (US$ M)	Remainder (US$ M)	LOM (US$ M)
Process plant**	575.06	446.68	-	1,022
Mining pre-production	10.78	-	-	10.78
Tailings and sediment	24.12	-	-	24.12
Pumping	2.34	-	-	2.34
Powerline	14.16	-	-	14.16
Road	2.59	-	-	2.59
Permitting and land acquisition	23.19	-	-	23.19
Mining sustaining	-	-	21.58	21.58
Other sustaining (including land permitting and land)	-	-	1.33	1.33
Closure	-	-	29.37	29.37
Total	**652.24**	**446.68**	**52.28**	**1,151**

***Includes contingency of 25% of plant direct costs.*

1.13 Operating expenditure estimate

A comprehensive and detailed operating cost estimate has been developed for the Project. It is summarised in Table 1-4.

Table 1-4 Opex estimate summary (pre-tax)

Area	LOM total (US$ M)	US$/t nickel**	US$/t ore	Average annual (US$ M)**
Mining	981	1,062	6.94	25.81
Rejects and tails handling	414	448	2.93	10.89
Processing costs	5,785	6,261	40.93	152.23
Royalties (CFEM)	23	25	0.16	0.60
Royalty (Vale)	66	72	0.47	1.74
G&A and other costs	215	233	1.52	5.67
HSE	24	26	0.17	0.63
Total	**7,508**	**8,126**	**53.13**	**197.57**

***Costs shown here represent an average over the LOM; actual costs for these components vary from year-to-year depending on the physicals.*

1.14 Market studies

1.14.1 Nickel and cobalt sulphate

In June 2019, HZM commissioned Wood Mackenzie to develop a report on the market for nickel and cobalt sulphate. It is summarised in Sections 19.1 and 19.2. As a consequence of that report, the following assumptions with respect to commodity pricing were used in the PFS:

- The consensus[1] nickel price of US$16,4000/t (US$7.44/lb) was used in the Base Case for the PFS along with a US$2,000/t (US$0.91/lb) premia for the production of nickel sulphate. The nickel sulphate premium is driven by the battery market (where nickel sulphate is valued higher than class 1 nickel) and is supported by very strong growth in the electric vehicle (EV) car market. The US$2,000/t (US$0.91/lb) sulphate premium is the average value for this for seen in the market over the last 12 months.
- The Wood Mackenzie long-term incentive price currently stands at approximately US$19,800/t (US$8.98/lb), this was used as an alternative case. A fixed price for nickel was applied over the LOM. The Qualified Person has reviewed the above and consider that the results support the assumptions in this Technical Report.
- The cobalt price assumption of US$34,000/t (US$15.43/lb) used in this study is significantly below the long-term consensus bank/broker forecasts which stand at US$55,000/t (US$25/lb).

1.14.2 Kieserite

In July 2019, HZM commissioned a report on the market for kieserite in Brazil from Dr Fabio Vale (Director Técnico/Technical Manager) of Adubai Consultoria Agronômica (Adubai). The report is summarised in Section 19.3. The report concludes that:

- The fertiliser market in Brazil is large. In 2018, 35.6 Mt of fertiliser was sold, of this 77.5% was imported and 22.5% was manufactured locally, The most likely consumers of the kieserite produced at the Project are the palm oil growers in Pará state, as palm oil trees have a very high demand for both magnesium and sulphur, although it has been demonstrated that coffee and cotton would also benefit from kieserite. The location of the Vermelho plant in the centre of the Pará state gives its distribution a competitive advantage over the imported product. The Project will produce approximately 150,000 t of kieserite a year, which is 10 times the current market for imported kieserite. This means there would be oversupply which would indicate a price reduction, and substitution of other agro-products would be required for all Project kieserite to be consumed. This indicates that it would be unlikely for current prices (approximately US$380/t FOB Barcarena) to be realised. For the study, HZM has assumed a

[1] CIBC

kieserite price of US$180/t (delivered) – about half of the current price in Barcarena. The study assumes a cost of US$80/t for delivery and marketing of kieserite, thus deriving US$100/t revenue.

1.15 Financial analysis

Table 1-5 and Table 1-6 show the Project headline economic results before and after taxation for a flat nickel price of:

- Base Case (consensus):
  - US$16,400/t for nickel + US$34,000/t for cobalt + US$100/t net revenue for kieserite.
- Wood Mackenzie (WM) long-term price:
  - US$19,800/t for nickel+ US$34,000/t for cobalt + US$100/t net revenue for kieserite.

Table 1-5 Project economic model headline results before taxation[2]

Item	Unit	Nickel price basis (US$/t Ni)	
		Base Case (consensus) 16,400	WM long-term price 19,800
Net cash flow	US$ M	10,379	14,655
NPV_8	US$ M	2,342	3,185
IRR	%	28.83%	34.48%

Table 1-6 Project economic model headline results after taxation[2]

Item	Unit	Nickel price basis (US$/t Ni)	
		Base Case (consensus) 16,400	WM long-term price 19,800
Net cash flow	US$ M	7,304	9,546
NPV_8	US$ M	1,722	2,373
IRR	%	26.26%	31.53%

1.16 Conclusions and recommendations

The Vermelho Project acquired by HZM is within the Carajás Mining District, which has well established mining infrastructure and services to support development of Vermelho. The previous Project owner had completed a FS to produce nickel and cobalt from open pit mining of the V1 and V2 deposits, processing these ores via the HPAL method. The project proposed in that FS was approved for construction.

In October 2018, HZM completed an FS on its Araguaia ferronickel project located which is located between 85 km and 150 km to the South of Vermelho. Consequently, HZM is familiar with working in the region and is well placed to consider what synergies exist in the development of the Araguaia and Vermelho projects.

MREs are deemed “current” and are reported in conformance with the CIM Definition Standards (2014).

Given the last drilling was completed 15 years ago, although to a high technical standard, combined with verification sampling completed as part of the HZM due diligence process, it is recommended to undertake a limited number of new drillholes to compare data against the historical drillhole data. This can likely be completed as part of any future metallurgical testwork program.

[2] Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t of the by-product, kieserite.

The MRE was completed by Vale 15 years ago, and there have been other partial model updates since, which have mainly focused on adjustments to the domaining scheme. The author recommends that the MRE is updated for the next phase of work including updating the naming of the lithological domains to bring it in line with the terminology utilised at HZM's Araguaia project.

The Project has been extensively studied and tested metallurgically, with LME grade nickel cathode produced with high recovery by the former Project owners, Vale using a beneficiation, HPAL, MHP ammonia re-leach and SX-EW process.

For this study, the current owners (HZM) elected to produce high-purity nickel and cobalt sulphate products suitable for the battery market in order to achieve a price premium. Simulus has selected the beneficiation, HPAL, MSP, POX, CoSX and crystallisation as the preferred process flowsheet which is proven at an industrial scale and is considered to be a more selective process (improved impurity rejection).

Extensive beneficiation and HPAL testwork completed by the former owners are directly applicable to the current Project. Samples used were representative of the LOM ores and were appropriately geographically spread over depth, ore types and lithologies.

The design criteria for the Project process plant is a peak ROM ore feed rate is 4.34 Mt/a at an average grade of 1.07% Ni. This is upgraded to 2 Mt/a at an average grade of 1.85% Ni based on beneficiation studies. HPAL metal extractions and acid consumption based on pilot testwork data are 97% for both nickel and cobalt at a 347 kg/t feed acid dose.

MSP produced during the pilot testwork achieved a grade of 54.6%, 3.22% and 37.3% for nickel, cobalt and sulphur, respectively. MSP impurities were low, with 0.4%, 3.6%, 0.24% and 40 ppm for aluminium, iron, silicon and manganese, respectively. This is suitable for feed to POX and subsequent CoSX and crystallisation to produce high-purity nickel and cobalt sulphate.

The POX, CoSX and crystallisation processing steps have not yet been demonstrated at pilot scale to reflect the current Project flowsheet; however, they are proven industrial processes. Therefore, an integrated pilot campaign with the selected flowsheet of beneficiation, HPAL, MSP, POX, CoSX and crystallisation is recommended during the next stage of Project development. The design nickel and cobalt recovery from HPAL feed to sulphate crystal product is 94.4% and 94.9% of the nickel and cobalt respectively.

Zero liquid discharge and dry stacking of leach residue solids have been included in the design to minimise environmental and safety concerns, so a kieserite ($MgS0_4.H_20$) by-product will be produced. The quality of commercial quantities of kieserite (nominally 158 kt/a) that is proposed as a by-product of the selected process, will require confirmation during pilot testing; however, it is expected to be suitable for sale to the fertiliser market and is expected to be relatively clean, as most of the heavy and base metal will be removed prior to its crystallisation. It is planned to be produced as a crystal dry product ready for bagging and trucking.

Mine design and schedules have been completed based on the MREs and on geotechnical data gathered by Vale. The geotechnical data should be updated in the next phase of development to reflect the current mining strategy.

Mining costs were developed from contractor submissions provided for HZM's Araguaia project in 2018 and escalated. In the next phase of development, these costs should be revised, and new submissions solicited that are specific to the Project and the current concept.

The previous EIA/ RIMA, LP and all environmental licences relating to the Vermelho mine and process plant contain background studies are relevant and will be useful to guide new social and environmental studies in the region. Once the concept design for the Project is finalised by the technical team within HZM, a new social and environmental impact assessment (EIA RIMA) should be conducted with the objective of obtaining an LP, and later LI to construct the Project in line with HZM's proposed Project characteristics.

1.17 Summary of the Project risks

1.17.1 Mining

Key mining risks include:

- Trafficability, wet weather and the related issue of selecting the mining equipment fleet. This is currently mitigated by lower mining rates in the wet season; however, further mitigation will be achieved when additional trafficability and geotechnical studies are completed.
- Predictability of grade when reclaiming from the large long-term stockpiles, which is mitigated by the mine to mill process.
- Mining contractor costs being materially higher to those predicted in this study.

1.17.2 Metallurgical

Overall, the metallurgical risks associated with the Project are considered to be low. The process flowsheet selected is based on industrially proven, conventional and robust unit operations including; HPAL, MSP, POX, CoSX and crystallisation as the main process steps. A review of the process design and available metallurgical testwork identified the following process risks:

- Lower nickel and cobalt grades in HPAL feed (due to beneficiation upgrade below that designed). This risk is considered low due to the extensive testwork completed.
- High acid consumption in HPAL. This risk was considered low, again due to the extensive testwork completed to date.
- Lower nickel and cobalt product purity than designed, preventing sale into the battery market. This risk was considered low to moderate due to the selected process route, but the status of current testwork has not yet demonstrated the level of purity required. The risk may be mitigated with further testwork in future stages of the Project development, which is recommended.
- Ability to sell the kieserite by-product, the purity of which has not yet been demonstrated via testwork. The risk may be mitigated after the recommended testwork is complete.
- Inability to achieve zero-liquid discharge, creating an unplanned effluent stream. This risk is considered low to moderate given the engineering completed to date. Possible risk mitigation to be evaluated includes; increased design margin for the kieserite crystallizer and diversion of excess steam from the acid plant (currently producing electricity) and/or installing effluent surge to even out short-term fluctuations in processing capacity.
- Poor stability of the HPAL residue solids from a geotechnical basis. This risk is considered low to moderate, based on operating examples around the world with similar leach residue solids using this technology. Testwork is recommended to confirm the properties of the solids and the dry stack tailings design. Possible risk mitigation includes blending with coarse beneficiation rejects and/or improved filtration or drying equipment.
- Low overall nickel and cobalt recoveries. This risk is considered low due to the engineering and testwork completed to date and the use of industry proven technologies.

1.17.3 Permitting and social

The permitting pathway for the Project is well defined and established. With several mines in the area in operation, permitting is not considered a high risk; however, there is a risk that the Project permitting process could be delayed due to competing priorities within agencies. This is being mitigated by engagement of regulators as HZM move the Project forward. Monthly reporting on permits and notifications should be continued.

The previously awarded (archived) licence for Vermelho contained a design to build a large tailings dam. This kind of layout would receive strong criticism today in light of recent tailings dam collapses in Brazil. It is important that HZM's team clearly communicate the dry-stack tailings method being applied by the Company and social/environmental impacts of this to agencies and communities.

If advanced, this would be the first HPAL nickel project to be licensed through to construction in Brazil and so HZM's team will need to support agencies by providing technical/expert advice and assist with benchmarking other HPAL projects around the world.

The social licence to operate is considered crucial to the Project's success. The community around the Project in general welcomes mining (as there are other operations in the area). However, there is a risk that community resistance may be encountered, such as a gap between those that work in mining and those that do not (inflation/housing availability). Accordingly, HZM will develop and implement plans to engage with the community during development and source local labour and suppliers for products and services where possible during operations.

The rural community living near the Project area must be engaged regularly to ensure ongoing positive relationships are maintained.

1.17.4 Market

The sensitivity study shows that the Project is most sensitive to the nickel price (and the sulphate premium). This is a high risk; however, the consensus pricing is higher than the Base Case nickel price applied in the cash flow model. Should the sulphate premium be less than expected, this will impact negatively on project economics.

The sensitivity study shows the Project is not sensitive to kieserite or cobalt prices, so the marketing risk for these products is considered a low.

1.18 Opportunities

1.18.1 Opportunities to increase the value of the Base Case

Mining

Direct tipping from mine to ROM hopper

It is envisaged that after production commences, the grade control processes put in place prior to mining will be viewed as sufficient to predict the mined grade (in some cases) and that material may be direct tipped from the mine face directly to the plant feed bin. By reducing this volume of rehandle, significant savings in mining costs will be realised.

Mining cost reductions

The FS-developed mining costs are based on proposals from contract mining operators in Brazil. These proposals were solicited using the mine schedule and with no negotiation, as there was no imminent contract. It has been recognised that when HZM is in a position to award a large mining contract, that negotiations should result in significant mining unit costs savings.

Processing

A review of the proposed process plant design, considering the selected flowsheet and products, resulted in the following list of possible opportunities to either reduce cost or improve the process:

- Revise acid plant configuration, current design is for two 1,200 t/d plants supplied by Outotec to match the planned project execution profile with the second stage being built in year 4. A single plant or alternative supplier will be cheaper.
- Investigate continuous screw presses rather than pressure filters in solid-liquid separation steps. The equipment is likely lower cost and associated footprint; ancillaries and infrastructure will also be cheaper.
- Use of self-cleaning, indirect heating on the autoclave feed and discharge rather than traditional direct contact heater vessels and flash cooling vessels. This is likely to result in a lower capital cost and reduced maintenance costs.
- Change from closed-circuit cooling water to an open-circuit cooling water system.
- Adopt the crusher equipment vendor’s suggestion that only primary and secondary crushing (roll crusher/sizer) is required, rather than the more conservative three-stage crushing that has been allowed for.
- Delete or defer the steam turbine generator to reduce initial capital (at the expense of some operating cost).
- Review opportunities for consolidation of reagents and utilities between the two HPAL stages to reduce capital costs.
- Review equipment suppliers for cheaper alternatives, particularly for high-cost items such as autoclave feed pumps, autoclave agitators, filter presses, air compressors and crystallizer packages.
- Investigate options to produce alternative nickel and cobalt battery material product such as ternary precursor which can trade at a premium to the nickel and cobalt content of approximately 140%.
- Investigate options to produce alternative products such as high-purity alumina and/or manganese metal, sulphate or dioxide, to add revenue streams for low marginal cost.

1.18.2 Opportunity to develop the Project as a ferronickel project

Early studies of Vermelho by Vale focused on smelting the saprolitic portion of Vermelho ore to ferronickel; however, this was later switched to a study of hydrometallurgical processing of the more limonitic ores (this forms the Base Case of this PFS). Based on laboratory testwork HZM carried out during the last quarter of 2018 and earlier work on HZM’s nearby Serra do Tapa deposit by the former project owners, as well as the close similarity of the chemistry of the blended feed over the first 10 and subsequent years to existing rotary kiln electric furnace (RKEF) operations, the RKEF process is also considered appropriate for the treatment of the Vermelho and Serra do Tapa saprolitic material; the ferronickel product would have a nickel content of 30% Ni.

Consequently, Snowden has developed a PFS case that explores the opportunity that utilises the Mineral Resources of the Project, in addition to HZM’s Mineral Resources at the nearby Serra do Tapa deposit, about 120 km by road from Vermelho (Figure 4-1), to feed a 1.8 Mt/a ferronickel plant using the RKEF process route to produce ferronickel.

This opportunity is discussed in more detail in Section 25.8.2. The post-tax economic key performance indicators (KPIs) are presented in Table 1-7 below.

Table 1-7 RKEF opportunity post-tax economic KPIs (excludes Fe credits)

Item	Unit	Nickel price basis (US$/t Ni)	
		16,400	19,800
Net cash flow	US$ M	5,327	7,657
NPV_8	US$ M	1,375	2,072
IRR	%	29.2%	36.8%

2 INTRODUCTION

2.1 Overview

This report is a NI 43-101 Technical Report on the PFS of the HZM wholly owned Vermelho Nickel-Cobalt Project. The Project is located in Canaã dos Carajás in the Carajás Mining District Brazilian state of Pará, approximately 760 km south of the state capital, Belém.

This report has been compiled by Snowden for HZM. HZM is the Project owner and is currently developing the Project. This report provides Mineral Resource and Mineral Reserve estimates and a classification of Mineral Resources and Mineral Reserves prepared in accordance with the CIM Definition Standards (2014). No Mineral Reserves have been estimated using Inferred Mineral Resources.

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

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

- Information available to Snowden at the time of compiling 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
- Data, reports, and other information supplied by HZM and other third-party sources.

The report contains capital and opex estimates with a combined accuracy of -20%/+25% with a confidence level of 50%.

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. The Qualified Persons for preparation of the report are:

- Anthony Finch, who visited the Project site on four occasions between June 2018 and September 2019
- Andrew Ross, who conducted site visits in December 2003 and in August 2019
- Simon Walsh, who has not visited the site.

The responsibilities of each author are provided in Table 2-1.

Table 2-1 Responsibilities of each co-author

Author	Employer	Employee title	Responsible for section(s)	Site visit	Site visit undertakings
Anthony Finch	Snowden	Executive Consultant	1, 2, 3, 4, 5, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27	On four occasions between June 2018 and September 2019	Review core at Vermelho, pit areas, landforms, road access, infrastructure, and local capability
Andrew Ross	Snowden	Executive Consultant	6, 7, 8, 9, 10, 11, 12, 14	December 2003; August 2019	The early visit served as an introduction to the geological setting of the deposits and operating procedures of Vale in respect of core drilling, surveying, logging, security, transport, subsampling and density measurements
Simon Walsh	Simulus	Principal Metallurgist	13, 17, 21.2.4, 21.3.5	No site visit performed; the focus of this Qualified Person was the plant flowsheet and process design, process capex and opex	

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

2.2 Issuer – Horizonte Minerals Plc

HZM is a nickel development company focused on Brazil which wholly owns the advanced Araguaia ferronickel project. HZM also has a 100% interest in the Vermelho Project (acquired from Vale S.A. in December 2017). Vale S.A. completed a full FS which demonstrated a nameplate capacity of 46,000 t of nickel per year and 2,500 t of cobalt per year. Vermelho is located in the Carajás region in the State of Pará, north-eastern Brazil, 70 km south of the Carajás region.

HZM was founded on 16 January 2006, with shares listed on the Alternative Investment Market of the London Stock Exchange (AIM) and on the Toronto Stock Exchange (TSX). The Company is incorporated and domiciled in the United Kingdom, with its registered office at Rex House, 4-12 Regent Street, London SW1Y 4RG.

As of 31 October 2019, HZM's institutional shareholder structure included Teck Resources Ltd, Canaccord Genuity Group, Richard Griffiths, Lombard Odier Asset Management (Europe) Ltd, JP Morgan, City Financial, Hargreaves Lansdown, and Glencore.

2.3 References

All references are listed in Section 27.

2.4 Abbreviations and units of measurement

Abbreviations and units of measurement are shown below.

Table 2-2 Abbreviations and units of measurement

Abbreviation/Unit of measurement	Description
%	percent
%w/w	percent weight by weight, or weight for weight
°	degrees
°C	degrees Celsius
100# or 100 mesh	100 openings per inch of screen, each opening is 149 micron wide
200# or 200 mesh	200 openings per inch of screen, each opening is 74 micron wide
3D	three-dimensional
AACE	Association for the Advancement of Cost Engineering
AAS	atomic absorption spectroscopy
Adubai	Adubai Consultoria Agronômica
Agoratek	Agoratek International
AIM	Alternative Investment Market of the London Stock Exchange
ANM	Departmento Nacional de Produção Mineral
AusIMM	Australasian Institute of Mining and Metallurgy
AR	ammonia re-leach
BEV	battery electric vehicle
BRL	Brazilian Real (plural: Brazilian Reais)
c.	circa
CAGR	compound annual growth rate
capex	capital expenditure
CFEM	Compensation for Exploration of Mineral Resources
CIM	Canadian Institute of Mining, Metallurgy and Petroleum

Abbreviation/Unit of measurement	Description
CIS	Cateté Intrusive Suite
cm	centimetre(s)
CMD	Carajás Mining District
Co	cobalt
COFINS	Contribution for the Financing of Social Security
CoSX	cobalt solvent extraction
CST	Construserv Servicos e Construcoes Ltda
Cu	copper
CVRD	Companhia Vale do Rio Doce
DDP	delivered duty paid
DFS	definitive feasibility study
DRC	Democratic Republic of Congo
EFC	Estrada de Ferro Carajás
Eh	Activity of electrons
EIR	Environmental Impact Report
EIS	Environmental Impact Study
EPCM	engineering, procurement, construction and management
ERM	Environmental Resources Management
EV	electric vehicle
EW	electrowinning
FFS	final feasibility study
FS	feasibility study
g	gram(s)
g/t	grams per tonne
G&A	general and administrative
Gamik	Vale’s internal laboratory
Golder	Golder Associates
GPS	global positioning system
ha	hectare(s)
HEV	hybrid electric vehicle
HDPE	high-density polyethylene
HPAL	high-pressure acid leach
hr	hour(s)
HZM (or “the Company”)	Horizonte Minerals Plc
ICMS	“Imposto Sobre Operações Relativas à Circulação de Mercadorias e Serviços de Transporte Interestadual de Intermunicipal e de Comunicações”; a state tax for goods and services
ICP-AES	inductively coupled plasma atomic emission spectroscopy
IFC	International Finance Corporation
IPI	“Imposto sobre Produtos Industrializados”. A federal VAT
IRR	internal rate of return
kbcm	kilo bank cubic metre (1,000 bcm)
kg	kilogram(s)
kℓ/a	kilolitres per annum
km	kilometre(s)

Abbreviation/Unit of measurement	Description
km^2	square kilometre(s)
kPa	kilopascal (1,000 pascals)
KPI	key performance indicator
kt	thousand tonnes
kt/a	thousand tonnes per annum
kV	kilovolt
kW	kilowatt
L&M	L&M Assessoria
LI	Installation Licence
Li-ion	lithium-ion
LME	London Metal Exchange
LO	Operation Licence
LOM	life of mine
LP	Preliminary Licence
m	metre(s)
M	million(s)
m^2	square metre(s)
m^3	cubic metres(s)
mbcm	million bank cubic metre (1,000,000 bcm)
MCP	Mixed Carbonate Precipitate
MHP	Mixed Hydroxide Precipitate
Miptec	Miptec Enginharia Consultoria
mm	millimetre(s)
Mm^3	million cubic metres
MRE	Mineral Resource estimate
MSP	Mixed Sulphide Precipitate
Mt	million tonnes
Mt/a	million tonnes per annum
MTO	material takeoff
mV	Millivolts
MVR	mechanical vapour re-compression
MW	Megawatt
NGO	non-government organisation
Ni	nickel
NI 43-101	National Instrument 43-101
NiEq	nickel equivalent
NiO	nickel oxide
NPV	net present value
NSR	net smelter royalty
opex	operating expenditure
PAE	Plano de Aproveitamento Econômico da Jazida
PAL	pressure acid leach
PCAs	“Planos de Controle Ambiental” or Environmental Control Plan
PEA	preliminary economic assessment

Abbreviation/Unit of measurement	Description
PFS	prefeasibility study
PGE	platinum group elements
pH	activity of hydrogen ions
PHEV	plug-in hybrid electric vehicle
PIS	Program of Social Integration
PLS	pregnant leach solution
POX	pressure oxidation leaching
ppm	parts per million
PPT	profit per tonne
QA	quality assurance
QAQC	quality assurance/quality control
QC	quality control
RC	reverse circulation
RF	revenue factor
RFP	Relatório Final Único de Pesquisa
RKEF	rotary kiln electric furnace
RFQ	request for budget quotation
ROM	run of mine
rpm	revolutions per minute
RPP	Relatório Parcial de Pesquisa
SAD69	Regional geodetic datum for South America in 1969
SEMAS	State Environmental Agency
SHFE	Shanghai Futures Exchange
Simulus	Simulus Pty Ltd
SM&A	Severino Macedo Consultoriae Engenharia
SMU	selective mining unit
Snowden	Snowden Mining Industry Consultants Pty Ltd
SRK	SRK Consulting
Steinweg	C. Steinweg Handelsveem (Latin America) S.A.
SX	solvent extraction
SX-EW	solvent extraction-electrowinning
t	tonne(s)
t/a	tonnes per annum
t/d	tonnes per day
TSX	Toronto Stock Exchange
TVR	thermo vapour re-compression
US$	United States dollars
UTM	Universal Transverse Mercator
Vale	Vale S.A.
VAT	value-added tax
VDS	Vale dos Sonhos
Vermelho (or “the Project”)	Vermelho Nickel-Cobalt Project
WALM	WALM Engenharia e Tecnologia Ambiental Ltda
WBS	work breakdown structure

Abbreviation/Unit of measurement	Description
WM	Wood Mackenzie
XRF	x-ray fluorescence
ZAG	Construtora Zag Ltda
Zn	zinc

3 RELIANCE ON OTHER EXPERTS

The author has relied upon the legal and environmental information provided by the following employees of HZM for inclusion in Section 4 (Property Description and Location):

- Mr Steven Heim, on 7 January 2018, provided an overview of mining legislation (Section 4.3) and tenement details (Section 4.3.2)
- Ms Katie Millar, on 3 January 2018, provided a status summary of environmental and social impact and other assessment permits (Sections 4.4 and 4.6).

In addition, Mr Bernardo Freitas of Freitas-Ferraz, the principal lawyer on the Vermelho acquisition for HZM, advised that the mineral rights phase is in between the first stage (Exploration) and the second stage (Mining) (Section 4.3.2).

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

Snowden and the Qualified Persons are reliant on HZM for the cash flow model estimates and results and the disclosed nickel price of US$16,400/t Ni with a US$2,000 premium for battery grade sulphate. The risks associated with the nickel pricing are disclosed and analysed in Section 24 in the price sensitivity discussion.

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

Information sources are shown in Table 3-1.

Table 3-1 Other parties relied upon to provide technical content and review

Information supplied	Other parties	Sections
Ownership, title, social and environmental studies and information	HZM	4
Infrastructure capital and opex estimates, logistics costs	WALM Brazil, Construserv Brazil, SM&A Brazil, C. Steinweg Handelsveem (Latin America) S.A.	18, 21, 22
Marketing report	Wood Mackenzie	19, 22
Taxation and royalties	L&M Assessoria	21, 22

4 PROPERTY DESCRIPTION AND LOCATION

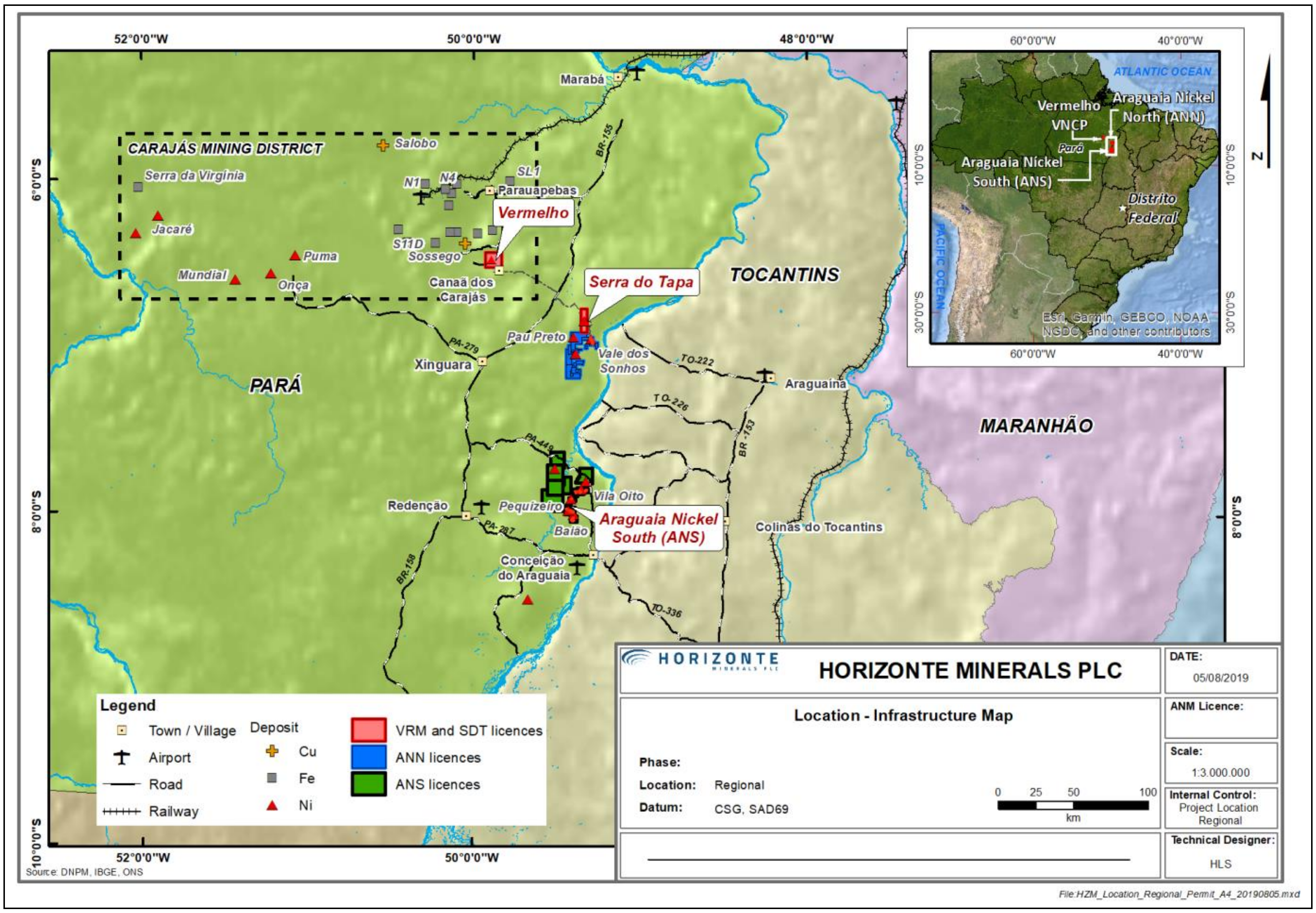
4.1 Introduction

The Project is located in the north-western Brazilian state of Pará in the Carajás region, approximately 760 km south of the state capital, Belém and approximately 85 km northwest of the companies Araguaia ferronickel project.

4.2 Location

The Vermelho licence is located near Canaã dos Carajás in the Carajás Mining District and 45 km south of the municipal district of Parauapebas in the Pará state, northern Brazil (Figure 4-1). Vermelho is located approximately 85 km from the northern part of HZM's Vale dos Sonhos and Serra do Tapa licence areas. Vale dos Sonhos is the northern part of the Company's Araguaia project.

Figure 4-1 Location map



Source: HZM, 2019

4.3 Licences and tenure

Brazil has a well-established permitting process for major mining projects, with a Mining Code and environmental legislation framework (CONAMA), which provides the support for companies to operate legally in the country.

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 authorisations, 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 "authorisation" 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 authorisation from the Federal Government. The exploration authorisation is granted through a licence issued by the Director General of the "Departmento Nacional de Produção Mineral" (ANM). ANM 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 ha 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 three years. The term (three years) can be renewed once, at the discretion of the ANM, upon its review of an interim Partial Exploration Report ("Relatório Parcial de Pesquisa" or 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-year period or in the case of renewal its second three-year period the holder must submit a "Relatório Final Único de Pesquisa" (RFP), the Final Exploration Report, on the results of the work to ANM. ANM may then decide to:

- Approve the report, when it shows the existence of a resource which can be both technically and financially developed
- Dismiss the report, when the exploration work undertaken was insufficient or due to technical deficiencies in the report
- File the report, when it has been proved that there was no deposit which may be both technically and/or financially developed, or
- 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.

The decision to postpone a decision on the Final Exploration Report is referred to as "Sobrestamento". With this decision ANM will fix a period in which the interested party will be required to submit a new technical-financial FS 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, ANM may grant the interested party an extension to the time limits or open a tender process for the licence if they believe there are third parties who could feasibly mine the deposit. If the new study demonstrates technical-economic feasibility, the RFP will be approved, and the holder of the licence will have one year to apply for a Mining Concession. An extension of one year can be requested in applying for the Mining Concession.

If the licence holder does not apply for the Mining Concession within the period mentioned above, the mineral rights over the property will lapse and the area becomes available for tender offers for 60 days, during which period any interested parties, including the previous licence holder, may submit their offers for an Exploration Licence or Mining Concession. The ANM will review the offers and will select the bid that, in its view, presents the most favourable conditions to meet the interests of the mineral sector. If no offers are submitted within the 60-day period, the area will then be considered as available for future applications for exploration licences under the priority system described above.

The application for a Mining Concession must be accompanied by the following information:

- I – The company's certificate of registration from the Board of Trade.

- II – Identification of mineral substances to be mined, with a copy of the exploration permit and the approval of the RFP.
- III – Name and description of the area intended for development, clearly and accurately reporting all river valleys or streams identified on maps or charts of known authenticity and exactitude; all railways and highways or any natural or topographical features of unmistakable determination; boundary lines with neighbouring Exploration Consents and Exploitation Consents if any; and the identification of the District, Municipality, Circuit Court District, and State; as well as the name and residential address of the owners or possessors of the land.
- IV – A graphic depiction of the intended area, circumscribed by a geometric figure formed by straight lines with a true north-south and east-west orientation, with two of their vertices, or in exceptional cases, one, anchored to a fixed, unmistakable point of the land, with the vectors defined by their lengths and true bearings, and showing the properties covered, indicating the names of the respective holders of rights to the surface of the soil, in addition to the site plan.
- V – Easements that apply to the mine.
- VI – Economic development plan of the deposit, with a description of the beneficiation plant.
- VII – Proof of the availability of funds or the existence of financial commitments necessary for the execution of the economic development plan and mining operations.

The economic development plan of the deposit known as the “Plano de Aproveitamento Econômico da Jazida” (PAE) consists of:

- I – A descriptive report.
- II – Pertinent designs or preliminary plans:
  - Mining method to be adopted, referring to the initially forecasted production scale and its projection
  - Lighting, ventilation, transportation, signalling and work safety plans in the case of underground mining
  - Surface transportation, beneficiation, and stockpiling of the ore
  - Power installations, water supply, and air conditioning
  - Hygiene of the mine and respective work
  - Housing facilities and their habitability for all who live in the mining area
  - Installations for the supply and protection of the origin, storage, distribution, and use of water for Class VIII deposits.

The design of the installations and equipment referred to in the economic development plan shall be consistent with the production justified in the descriptive report and include a forecast for future expansions.

The Mining Concession will be denied if the development is considered by the Government to be prejudicial to the public welfare or compromises interests that transcend the use of the industrial exploitation. In the latter case, the explorer shall have the right to receive indemnification from the Government for expenses spent on the exploration work, once the exploration final report has been approved.

The holder of a mining concession must inter alia: (i) commence development within 180 days from the granting of the concession, subject to obtaining all required environmental licences and authorisations; (ii) refrain from suspending development and mining operations for more than six months without the prior approval of the ANM; (iii) mine according to the mining plan approved by the ANM; (iv) compensate the land owner for occupation of the property; (v) pay a royalty to the landowner; (vi) pay a royalty to be distributed among the local, state and federal governments; (vii) obtain all required environmental licences and authorisations; (viii) restore the areas degraded by mining and processing operations and infrastructure; (ix) report annually to the ANM on activities, production and sales.

Mining concessions may be transferred (in whole or in part) to legal entities incorporated in Brazil, if the transferee demonstrates technical and financial capability to the ANM. The transfer is subject to the approval of and registration by the ANM. Furthermore, mining concessions can also be encumbered, e.g. because of a judicial order or as a security. The mining concession may be relinquished by its holder at any time. In such event, the holder will, at the discretion of ANM, be able to remove its property from the mine location provided that no damage is caused to the mine.

In general, mining projects must undergo a three-stage environmental licensing process. Generally, the State environmental authority oversees licensing a mining project for projects contained within one State, as opposed to the Federal environmental authority (IBAMA) whom are responsible for licensing mining projects across state borders. The Federal environmental authority will be in charge whenever mining activities will be undertaken in, or cause an impact on, areas deemed as federal, such as national environmental conservation units, as well as in cases where mining activities will be executed in two or more States. The compliance with all applicable environmental laws includes, but is not limited to, the possession by mining companies of all permits and other governmental authorisations required under applicable environmental laws, and compliance with the terms and conditions thereof, including the authorisations granted to impound water and exploit forest resources.

A Preliminary Licence (“LP”) must be obtained at the planning stage of the mining project. A Social and Environment Impact Assessment (“EIA RIMA”) and a plan for the restoration of degraded areas must be prepared at this stage. Public hearings are usually called, to present the EIA RIMA to the communities and authorities. Following the public hearing, the State Environmental Agency (“SEMAS”) may or may not approve the issuing of the LP. The LP usually imposes conditions that must be complied with by the mining company. 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.

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.

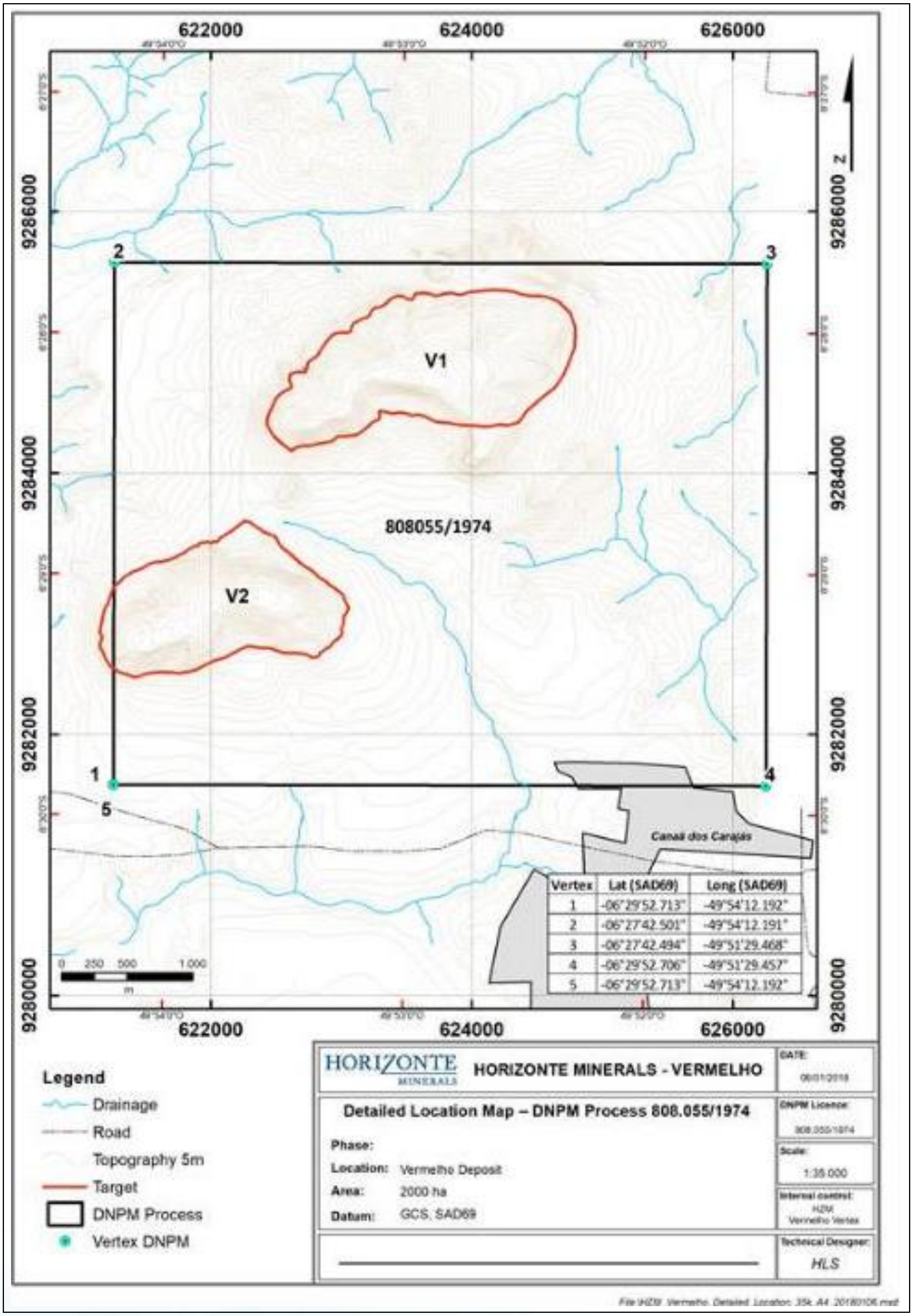
The last stage of the environmental licensing process is the one related to the Operation Licence (“LO”). The LO is granted once the environmental authorities are satisfied that the development and construction were completed in accordance with all the conditions of the LI and that the PCA is correctly implemented. The LO authorises 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. It is possible to renew the LO if the request is presented 120 days (or another period set by specific regulations) before the expiry date of the last permit. In that case, the LO is automatically extended until the environmental agency discloses its final decision about the request.

4.3.2 Vermelho licence

On 28 June 1974, Vale filed a request for an Exploration Licence which initiated ANM Process 808.055/1974. Exploration Licence 1111 was published on 3 April 1978. At the end of the initial three-year exploration period, Vale requested an extension which was renewed under the new licence number 1727, published on 12 June 1981. The Vermelho licence is described by ANM Process 808.055/1974 and covers 2,000 ha, including the V1 and V2 nickel-cobalt laterite deposits (Figure 4-2). The title was transferred from Vale S.A. to Vale Metais Basicos S.A. in 2015.

Currently, the Mineral Rights process is at the “Plano de Aproveitamento Econômico da Jazida” (economic development plan or “PAE”) stage which was submitted by HZM in March 2019.

Figure 4-2 Licence surveyed coordinates



Source: HZM, 2019

In Table 4-1 (below), all licences are in good standing. The date shown in column 5 represent the most recent ANM reporting or other action to be taken for each process.

Table 4-1 Licences relevant to the Project and the current status

Process ID	Area (ha)	DOU publication	Process stage	Current status	Action option
808.055/1974	2,000	4 March 1978	5	PAE + Requerimento de lavra filed 06/03/2019	PAE approval monitoring

HZM has not acquired any surface land rights for the Project, but the Company has agreements in place with the principal landowners for surface access rights covering the main deposits.

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, and progress to the construction licence stage.

4.5 Environmental obligations

The community of Canaã dos Carajás located on the southern boundary of Vermelho has seen significant development resulting from mining projects in recent years. The population increased almost tenfold from approximately 4,000 in the 1990s to 36,000 in 2017 (IBGE, 2017).

The Project does not include any publicly registered Indigenous reserves or federal/state forests.

The main economic activities for the population include mining (copper and iron) and cattle farming.

A Full Environmental and Social Impact Assessment (EIA RIMA) was conducted and the Preliminary Licence (LP) subsequently granted for the Project, demonstrating the government's approval of the environmental and social viability for the proposed HPAL project to produce 46,000 t/a nickel in cathode, 2,800 t/a cobalt and 500 t/a of copper (by-product) in the Canaã dos Carajás region. A new licence process will be undertaken for Vermelho as detailed in Section 20 of this report.

HZM will maintain 80% of any lands controlled (acquired/rented/other) as a native legal reserve (native vegetation) as required by Brazilian environmental legislation for lands located in the Amazon biome.

4.6 Permits

The permitting pathway was advanced by Vale for the Project, and is summarised in the following:

- Full Environmental and Social Impact Assessment (EIA RIMA) completed by CEMA consultants
- Preliminary Licence (LP) for Vermelho was granted to Vale in 2006, demonstrating government approval of social and environmental viability of project proposal
- Vale submitted a request for the Installation Licence (LI) in May 2007
- Vale decided not to progress the Project, and the environmental licence process was discontinued archived with SEMAS as a result.

The current Project status is summarised as:

- The previous EIA RIMA, LP and all environmental licences relating to Vermelho mine and process plant contain background studies that will be useful to guide new social and environmental studies in the region.

- Once the concept design for the project is determined by the technical team within HZM, a new EIA RIMA will be conducted with the objective of obtaining an LP, and later LI to construct the Project in line with HZM's proposed Project characteristics.
- HZM will require engagement with community groups in the region, as well as authorities – such as, but not limited to:
  - SEMAS – Pará State Environmental Licensing agency
  - INCRA – Brazilian Land Authority
  - ITERPA – Pará State Land Authority
  - IPHAN – Brazilian Institute of Historic and Artistic Heritage
  - Municipal level authorisations.

The permitting pathway for Vermelho is detailed in Section 20 of this report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, elevation and vegetation

As described in AMEC (2006), the Project area is characterised by relatively flat terrain, with two hills forming the V1 and V2 deposits (Figure 5-1). The hydrographical basin is formed by the Araras and the Verde rivers, both of which are tributaries of the Parauapebas River.

The Project is located in the Amazon biome. The original vegetation is the equatorial latifoliated forest, with transitions to a tropical forest, dominated by low and medium size plants, and locally with very tall trees, including the cedar (*Cedera odera*), ipê roxo (*Tabebuia barbata*), angelim (*Hymenolobium petraeum*), maçaranduba (*Manikalra luberi*). Other common species include the cipós and various species of palm trees. Several decades ago, most of the area has been deforested, and is currently used for agriculture.

Figure 5-1 Topography and vegetation

Source: AMEC, 2006

5.2 Access

Access to the area is by commercial flight from Brasilia, to Marabá (Pará), then by road from Marabá (by highway BA-155 and PA-275), along 227 km of paved road to Canaã dos Carajás. Alternatively, access is possible by commercial flight from Belém (Pará) to Canaã dos Carajás. The licence area is accessible in a 4x4 vehicle along farm/drill tracks.

5.3 Proximity to population centre and transport

The nickel laterite deposits are 6 km northeast from Canaã dos Carajás which was founded during the 1980s. The 2004 census recorded a district population of 13,000, mostly occupied with agriculture, cattle farming, and related services. The population increased approximately 30% from 26,700 in 2010 to approximately 35,000 in 2016.

Mineral discoveries in the region (copper at Sossego, nickel at Canaã dos Carajás) boosted the interest in the area. Local living and working conditions are improved due to major investments from Vale.

The regional transport is good; railroads and highways connect the towns and cities of the state. Regularly scheduled air services are available in Carajás and Marabá, a main urban centre in this region, approximately 180 km from Carajás by highway. Most flights connect to Brasilia and to Belém, the capital of Pará. Parauapebas and Marabá are connected by the Estrada de Ferro Carajás (EFC) railway to São Luís and by the PA-275 and PA-150 highways to Belém.

5.4 Climate and length of operating season

The Vermelho property is in the Amazonas sedimentary basin. The climate is characterised by a hot and humid tropical climate, with two distinct seasons. A rainy season occurs from November through April, and a dry season from May through October. During the 30-year period from 1968 to 1998, the annual rainfall averaged 1,930 mm. On average, the heaviest rainfall is generally in March. Temperatures range from a low of 15°C to a high of 35°C.

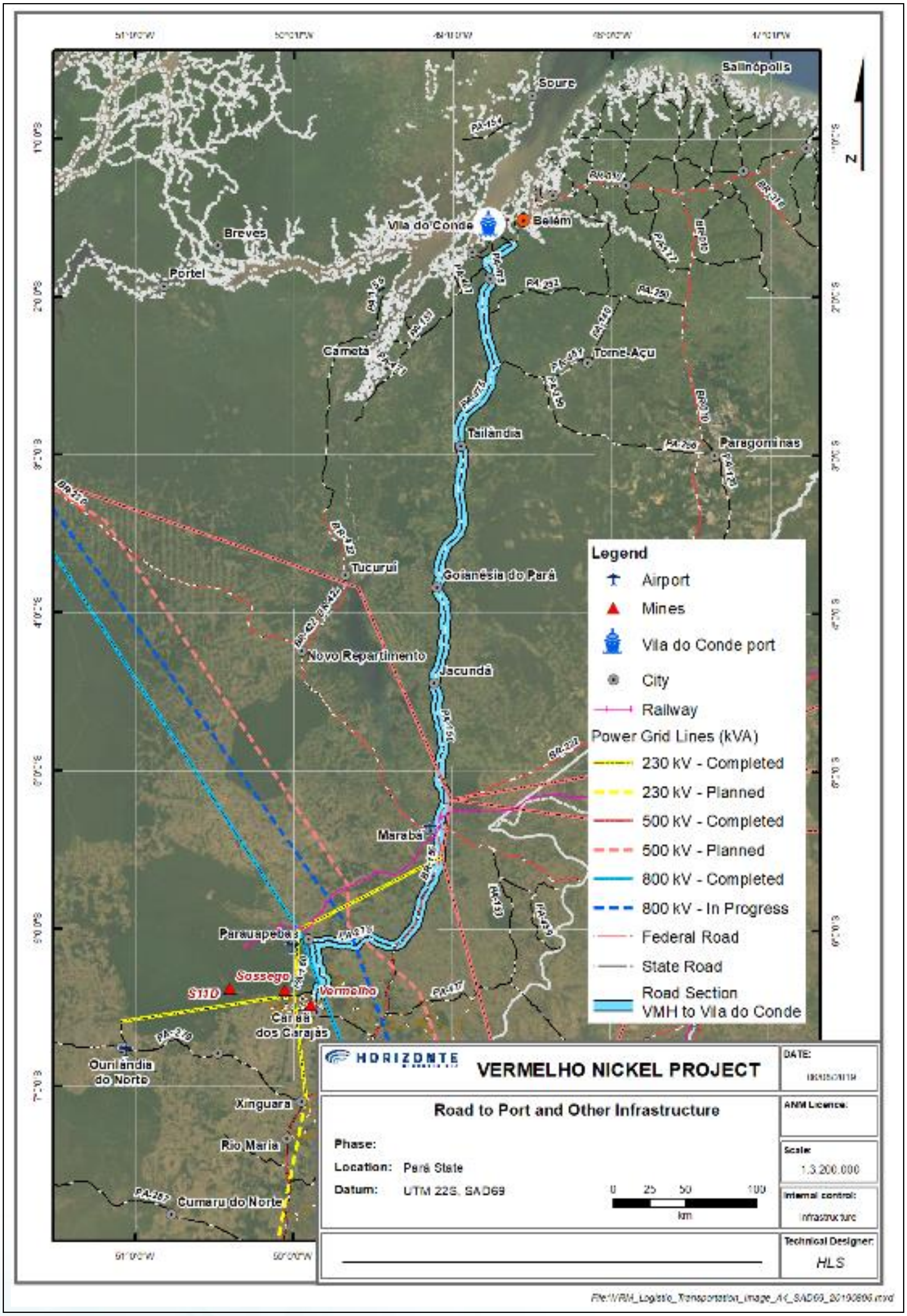
5.5 Surface rights

HZM has not acquired any surface land rights for the Project but the Company has agreements in place with the principal 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 suitable terms cannot be agreed between the landowner and company. HZM currently has good working relationships with the principal landowners.

5.6 Infrastructure

Infrastructure is shown in Figure 5-2. the Carajás district is an established mining region with well-developed infrastructure in place, including rail, roads and hydro-electric power.

Figure 5-2 Infrastructure



Source: HZM, 2019

6 HISTORY

6.1 Previous exploration and development work

The Vermelho area was explored in various stages by DOCEGEO, the exploration company of Vale. From 1974 to 1976, the presence of ultramafic rocks south of Serra dos Carajás were identified through the interpretation of radar data and photo-geological surveys, and subsequently followed by ground exploration, soil sampling, and limited drilling and excavation of shallow pits.

Further exploration was conducted in 1980 to 1982 and again from 1988 to 1991, comprising the digging of shallow pits, which allowed a better assessment of the nickel potential. More detailed exploration campaigns were conducted from 1993 to 1994 and 1997 to 1998, including drilling, shallow pits, and detailed sampling. By 1997, the exploration and metallurgical work indicated that a High-Pressure Acid Leach (HPAL) route could be feasible. After this period, the total historical resource was estimated by Vale at 32.9 Mt of garnieritic material averaging 1.7% Ni, and 190.9 Mt of lateritic material that graded 0.93% Ni and 0.08% Co. The drilling grid density was substantially enhanced, and most of the resources were upgraded to the Measured category. Pilot plant metallurgical studies were conducted in Australia. A PFS was prepared in 2003, and a FS was completed in August 2004 by GRD-Minproc (2005). This study confirmed the positive economic outcomes obtained in previous studies and showed production capacity of 46,000 t/a of metallic nickel, and 2,500 t/a of metallic cobalt. The Project was given construction approval by the Vale board in 2005. Later that year Vale elected to place the Project on hold after it acquired Canadian nickel producer, Inco.

Approximately 152,000 m of combined drilling and pitting was completed by Vale (Table 6-1).

Table 6-1 Summary of Vale exploration

Description	V1		V2		Project area		Total	
	Unit	Drilling (m)	Unit	Drilling (m)	Unit	Drilling (m)	Unit	Drilling (m)
Pit excavation	251	3,903	231	3,585			482	7,487
Large-diameter drilling	86	3,582	152	7,333			238	10,915
Diamond drilling	39	2,931	24	1,806			63	4,736
Rotary percussion	1,289	72,875	867	50,768	65	1,370	2,221	125,013
Exploration pits					11	60	11	60
Mixed geotechnical drilling					20	788	20	788
Percussive geotechnical drilling					75	783	75	783
Auger drilling	15	8	13	8	296	265	324	280
Rotary geotechnical drilling	9	945	9	906			18	1,851
Totals	**1,689**	**84,243**	**1,296**	**64,405**	**467**	**3,265**	**3,452**	**151,913**

Source: Extracted from Vale, 2004

6.2 Historical Mineral Resource and Mineral Reserve estimates

6.2.1 Historical Mineral Resource estimates (MREs)

Following review by the author in February 2018, the sequence of MRE studies over the period 2003 to 2008 is shown in Table 6-2 below.

Table 6-2 Sequence of MREs

Model name	Generated by	Year	Block size	Software	Audited by
PFS_25_2m	Vale	2003	25 x 25 x 2 m	Gemcom	Snowden
Updated PFS_25_2m	Vale	2004	25 x 25 x 2 m	Gemcom	Snowden
FFS_25_2m	Vale	2004	25 x 25 x 2 m	Gemcom	Snowden
RFS_12.5_3m	Vale	2005	12.5 x 12.5 x 3 m	Gemcom	AMEC
CVRD-SRK_12.5_3m	Vale-SRK	2007	12.5 x 12.5 x 3 m	Gemcom	Not done
Snowden_25_1m	Snowden	2008	25 x 25 x 1 m	Datamine	Incomplete

The author reviewed these MREs and concluded that the “FFS_25_2m” MRE and block models were appropriate for use by HZM. Details are provided in Section 14.

6.2.2 Historical Mineral Reserve estimates

There are no historical Mineral Reserve estimates to report in this Technical Report. Current Mineral Reserves are reported in Section 15.

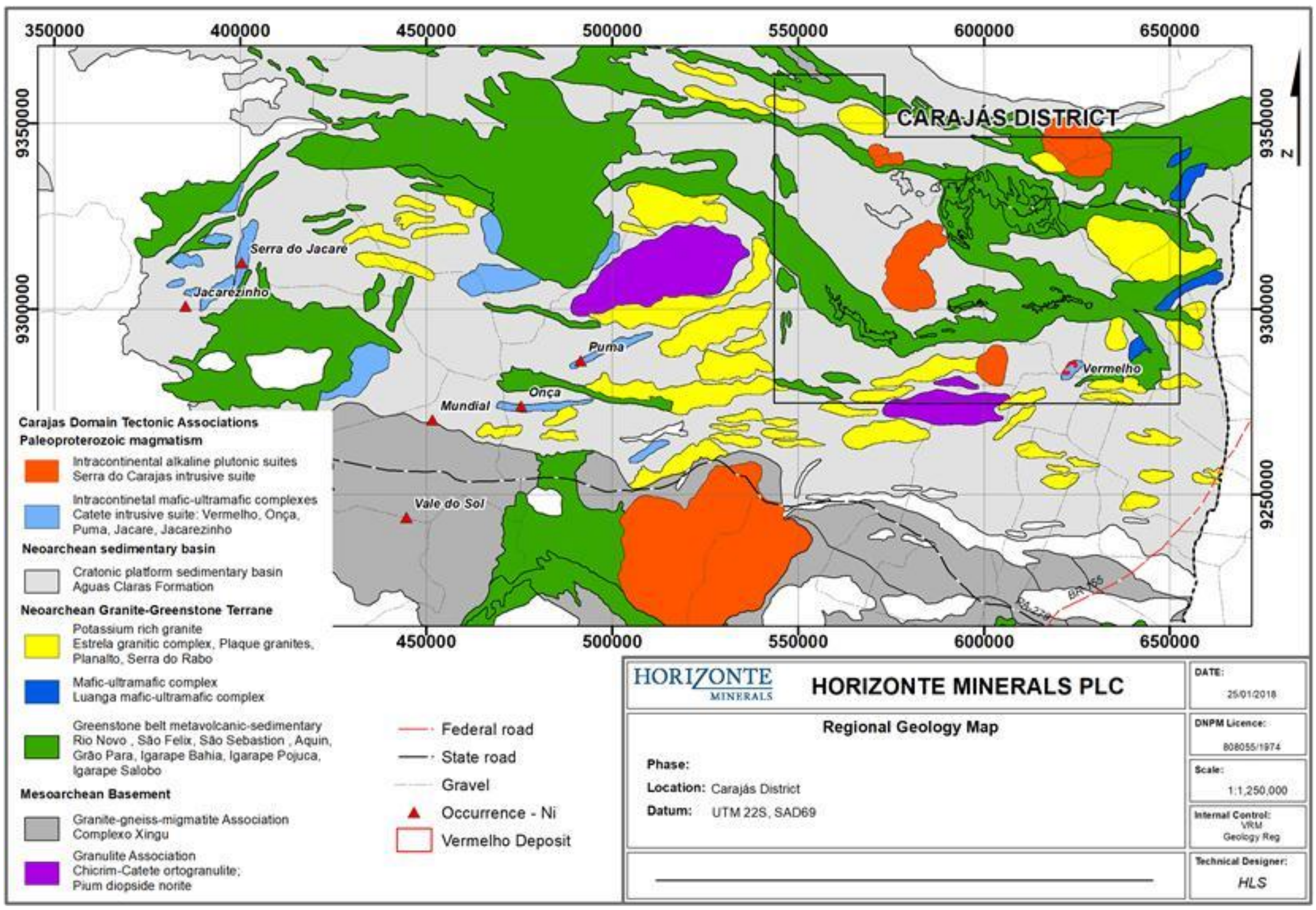
7 GEOLOGICAL SETTING AND MINERALISATION

The following is extracted from Vale (2003), Vale (2004), and AMEC (2006) reviewed and amended by the author for inclusion in this Technical Report.

7.1 Regional geology

The Vermelho Nickel-Cobalt Project is in the south-eastern portion of the Carajás Mineral Province. The deposits have formed because of supergene weathering and leaching of mafic and ultramafic rocks of the Vermelho layered intrusion, which belong to the Cateté Intrusive Suite (CIS). These are anorogenic bodies, intrusive in Archean rocks of the Xingu Complex (migmatites, granite-gneisses) and have been dated at 2.4 Ga. They have not been deformed or metamorphosed and their intrusion is related to distensional Palaeoproterozoic events. Other well-known intrusive bodies belonging to the CIS are Serra da Onça, Serra do Puma, Igarapé Carapanã, Serra do Jacaré, Serra do Jacarezinho, Fazenda Maginco and Ourilândia bodies, distributed from east to west in the southern portion of the Carajás Mineral Province (Figure 7-1).

Figure 7-1 Regional geological setting



Source: HZM, 2019

The magmatic fractionation of primitive magmas hosted in a sialic continental crust has generated a series of multi-layered mafic-ultramafic bodies outcropping in the central-eastern portion of the Carajás Mineral Province.

They are mafic-ultramafic bodies, layered, undeformed or metamorphosed, oriented according to the structural trends of the Itacaiúnas Belt, composed from the bottom to the top by serpentinites, peridotites, pyroxenites and gabbroic rocks. These characteristics indicate an inter-continental plate environment intruded in the rocks of the Xingu Complex, Granite Plaquê and São Félix Group, in an anorogenic extensional regime, presenting metallogenetic potential for platinum group elements (PGEs), nickel-copper sulphides and chromite. The magmatic stratigraphy of the bodies and their primary structures show an evolution in dynamic magmatic chambers from various magmatic pulses.

The Vermelho mafic-ultramafic complex is located about 20 km southwest of Serra do Rabo and is characterised by serpentinites originating from a dunitic to peridotitic protolith, from which a nickeliferous lateritic profile originated.

7.2 Deposit geology

The Vermelho nickel deposits consist of two hills named V1 and V2 (after Vermelho 1 and Vermelho 2), aligned on a northeast-southwest trend, overlying ultramafic bodies. The ultramafic bodies have had an extensive history of tropical weathering, which has produced a thick lateritic profile of nickel-enriched limonite and saprolite at V1 and V2.

The V1 and V2 deposits form flat lying topographical highs (refer to Figure 5-1), where V1 body reaches heights of around 500 m, standing out from the adjacent terrain about 175 m whereas the V2 body has a maximum height of 450 m. The flattened terrain adjacent to the two bodies represents the gneiss ground that hosts the layered intrusion of Vermelho. The V2 hill is located approximately 2 km southwest of V1. The V1 hill has a deformed convex-concave shape (convex to northeast), and extends for approximately 2.4 km east-west, ranging from 700 m to 1.6 km north-south. The V2 hill has an east-west elongation, and extends for approximately 1.9 km east-west, ranging from 600 m to 900 m north-south. The V1 deposit has an average thickness of 53 m and a maximum thickness of 146 m, whereas the V2 deposit has an average thickness of 56 m and a maximum thickness of 115 m.

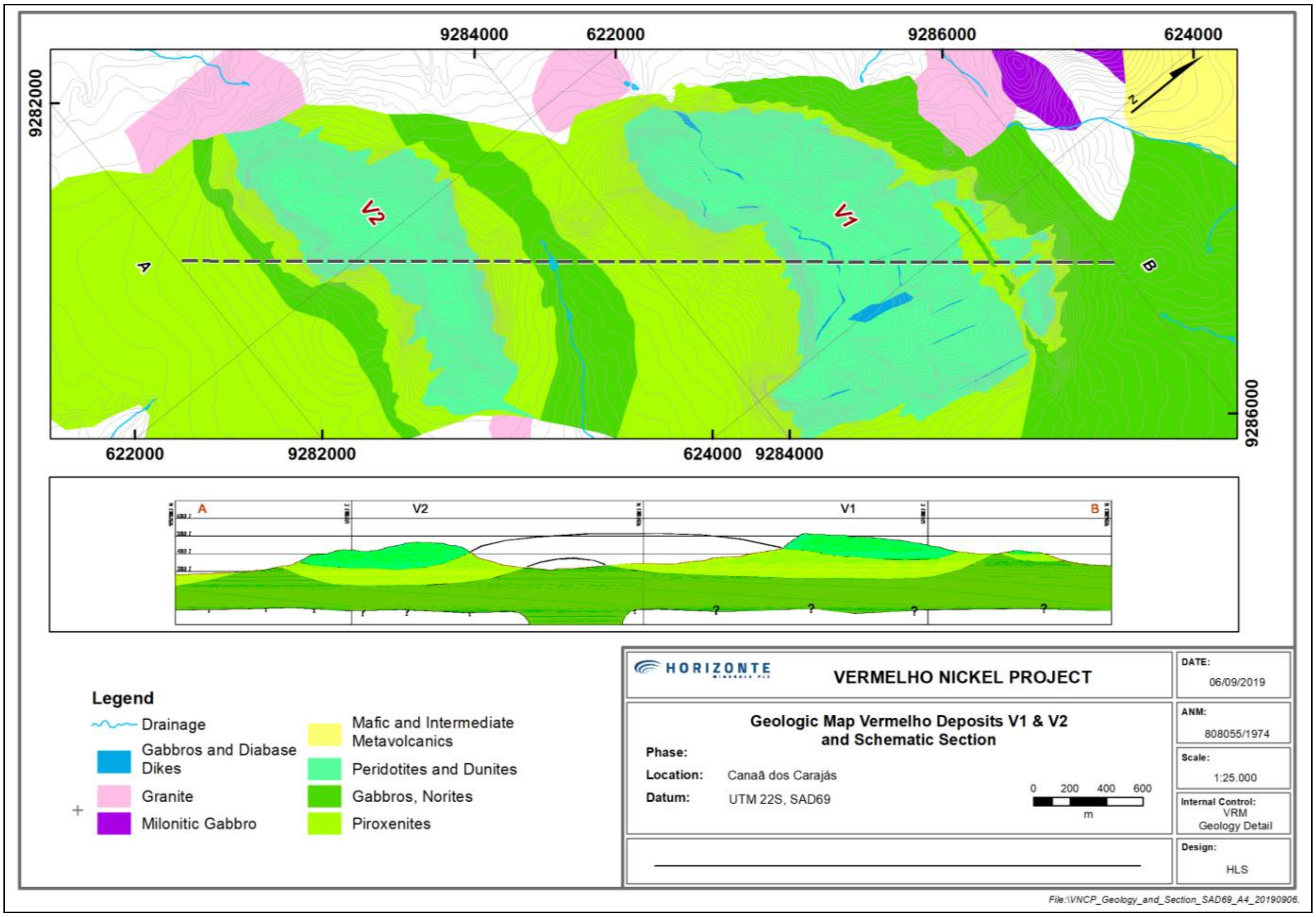
The ultramafic bodies are erosional relicts of the upper sheet of a three-layer intrusion represented, from bottom to top, by a mafic zone (gabbros, gabbro-norites and leuconorites), a pyroxenitic zone (orthopyroxenites and chromitites, with c. 50 m thickness) and a peridotitic zone (serpentinised dunites and harzburgites, with c. 150 m thickness). The northeast-southwest oriented, 10 km long and 2 km wide, layered intrusion has intruded a package of gneisses and migmatites belonging to the Xingu Complex. Late sub-vertical diabase dykes intersect the three layers in different directions. Various chromitite levels have been identified at the southern sides of both V1 and V2 within the pyroxenite zone.

7.3 Lithologies

The framework upon which the Vermelho nickel deposit has developed consists of three sub-horizontal mafic-ultramafic rock layers shallowly dipping north with an approximate 10 km long, N45E-oriented 2 km wide strip, with the following distribution from bottom to top (Figure 7-2):

- Mafic Zone, (unknown thickness) comprising very heterogeneous silica-rich lithotypes, mainly gabbros, gabronorites and norites. It is a heterogeneous zone, with compositions ranging from mafic to intermediate. Gabbros of intermediate composition have interstitial quartz and zircon.
- Pyroxenitic Zone, (approximately 50 m thick) comprising orthopyroxenites and layers of cumulates chromitite. Primary mineralogy can be well preserved, with localised transformations for talc, serpentine, magnetite and carbonate. Massive chromitites (cumulated chromite) exhibit adcumulated texture.
- Peridotitic Zone, (approximately 150 m thick) comprising serpentinites (dunites and harzburgites serpentinised). The analysed samples did not present the preserved magmatic silicates, although the primary rock textures are well preserved. Dunites have adcumulatic textures (cumulated olivine) and harzburgites mesocumulatic textures (olivine cumulated with orthopyroxene intercumulus).

Figure 7-2 Deposit geology with cross section



Source: HZM, 2019

The contact between the Pyroxenitic Zone and the Peridotitic Zone is a consistent east-west orientation dipping gently north (Figure 7-3).

Figure 7-3 Contact between Peridotitic Zone (magenta) with Pyroxenitic Zone (yellowish)

Source: HZM, 2019

The fine to medium granulated serpentinite presented in the Peridotitic Zone is a product of hypogene alteration of the dunites and peridotites. It is a dark green to black rock with density around 2.5 (Figure 7-4, E).

Magmatic levels with 1–10 cm of chromitites can be found in V1 and V2 hills within the Pyroxenitic Zone (Figure 7-4, C and D). There are massive layers of Fe-chromite, magnetic with the primary texture preserved, but with no Precious Group Metals.

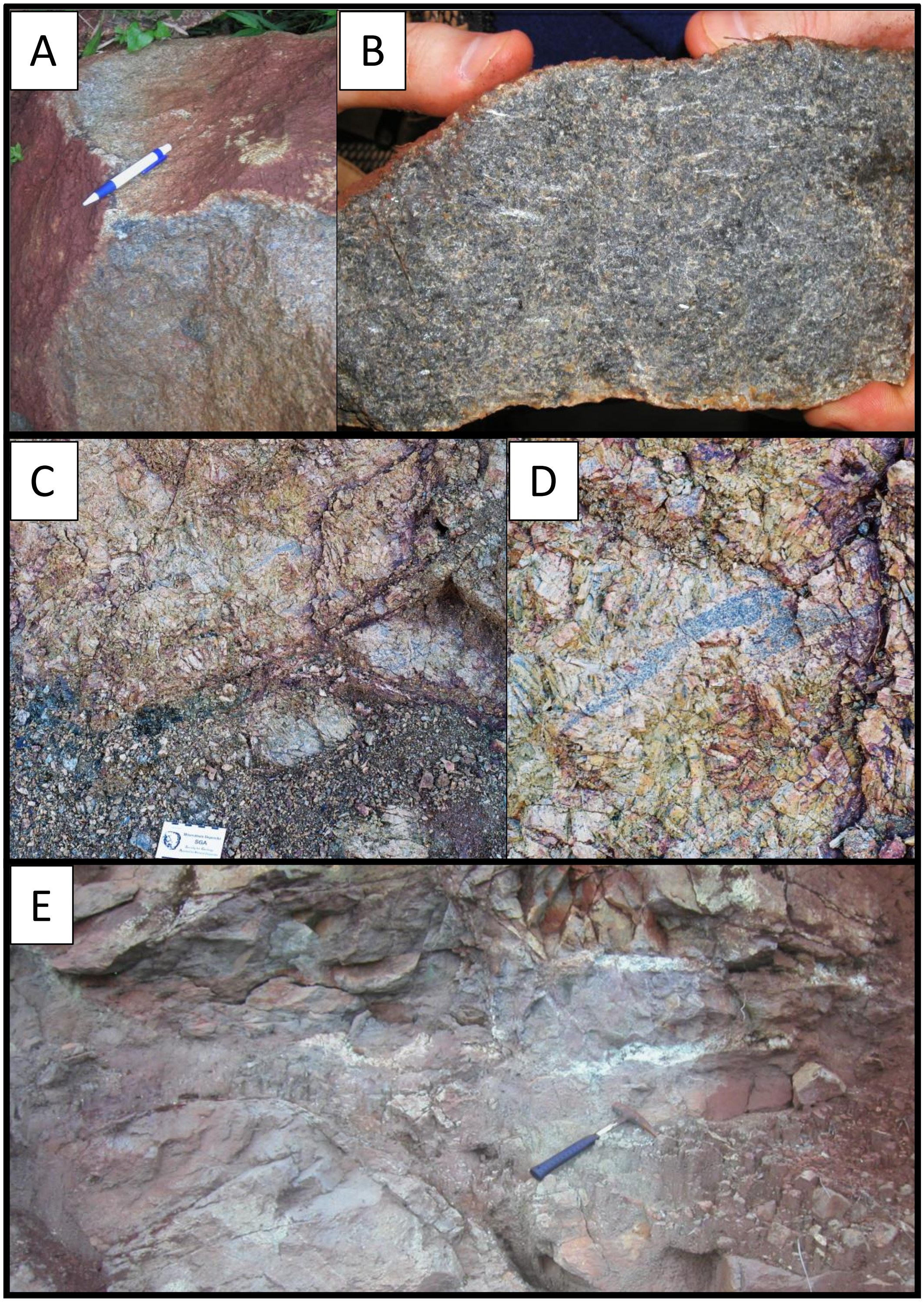
The Mafic Zone occurs in the northern, southern and central massifs and they are formed by gabbroic rocks (Figure 7-4, A and B). The pyroxenite lenticular intercalations are common in the serpentinitic body. Generally, the mafic rocks correspond to zones of low relief. Mafic dykes, interpreted as diabasic bodies, cut through the whole sequence, especially in V1.

The sequence has undergone hydration as the magmatic chamber aged. Dunites and peridotites are strongly serpentinised, and pyroxenites are amphibolitised. The silicification which supports and preserves the topographic relief is the starting point of the weathering processes.

The arrangement of the Vermelho layered Intrusion lithotypes, where the primitive ultramafic cumulates (dunites, harzburgites) sequentially situated below the orthopyroxenites and mafic cumulates, indicates that the rocks have become progressively more primitive from bottom to top. This may be explained by either: (a) the stratigraphy has been tectonically inverted or (b) the Pyroxenitic and Mafic zones are associated with the border of a magma chamber and the Peridotitic Zone represents the bottom of another large magma chamber, starting a new cycle, now eroded.

The first alternative is unlikely because of absence of tectonic structures that justify the inversion of stratigraphy.

Figure 7-4 Lithologies



Source: HZM, 2019

Notes:
- *A – Mafic zone outcrop located at V2.*
- *B – detail of a leucograbbro with orientated plagioclase crystals at V1.*
- *C – Pyroxenitic zone outcrop located at V1.*
- *D – Detail of an irregular chromite layer.*
- *E – Weathered dunite (peridotitic zone) at V1.*

7.4 Structural features

Structural data analysis was based on measurements collected in detailed geological mapping (at a scale of 1:2500) on V1 and V2 by Vale. Analysis completed by Vale shows that both the igneous foliation and the tectonic foliation occur as parallel-to-sub-parallel layers, orientated horizontally-to-sub-horizontally in a slightly undulating manner. This structural regime assists the conclusion that the deposit did not undergo intense deformations throughout the geologic evolution of the region. The magmatic character in layered intrusion remains even after later tectonic events that could have deformed the complex.

Joints and fractures constitute the most significant structures in bodies V1 and V2. Two main groups of joints and fractures were identified, interpreted as possible conjugate pairs, with orthogonal northeast/northwest and north-south/east-west directions.

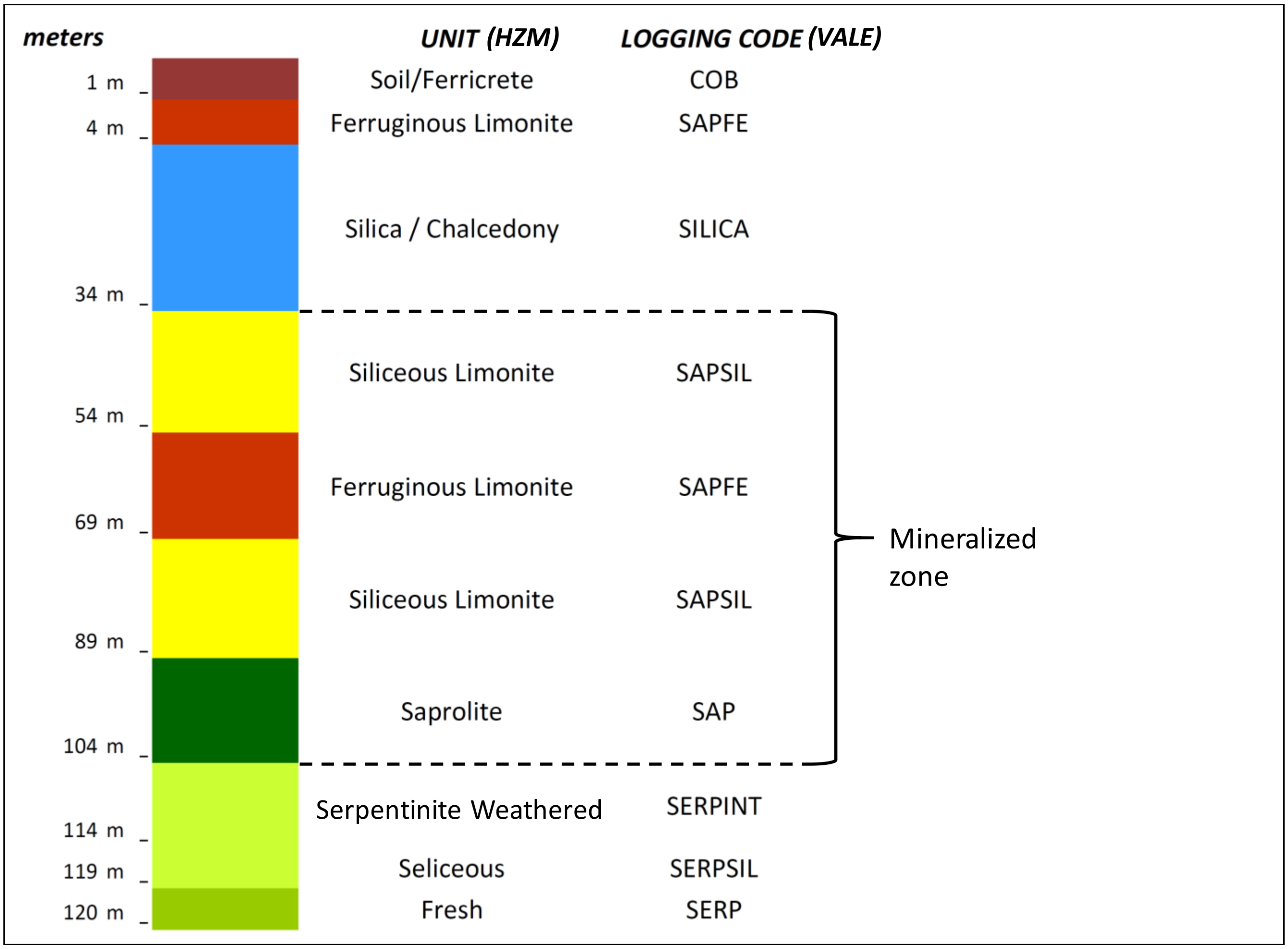
The structural features at the deposits are interpreted to be fundamental to the evolution of the weathering and consequently to the mineralisation. The horizontal to sub-horizontal foliation planes in association with the vertical joints and fractures worked as conduits for water migration during weathering and assisted the leaching processes and the precipitation of silica and other elements responsible for the formation of the boxwork texture presented in the silicified saprolite ore.

7.5 Weathering profile

The technical literature uses many names to classify lateritic weathering profiles, depending on when it was classified, where and by whom. In general, nickel laterites have a superficial layer of soil, where sometimes duricrust can appear, a limonitic zone, a saprolitic zone and bedrock. There are cases (e.g. HZM's Araguaia project) where a mixture of limonite and saprolite occurs, which is classified as transitional. Despite the work that Vale carried out in the PEA, PFS and FFS on the project, the HZM technical team elected to update the nomenclature of the weathering units so that they are in alignment with HZM's other projects. Consequently, a typical weathering profile with the logging codes used in the past by Vale and its updated HZM unit names is presented in Figure 7-9.

As previously described, the weathering evolution of the Vermelho Mafic-Ultramafic Complex resulted in a truncation of lateritic evolution and the resumption of the process after climate change. The formation of lateritic soil was followed by significant silicification, generating the thick layer of chalcedony, erosion of the primary lateritic package and generation of new lateritic profile under the silicified level. Thus, the profile presents interdigitations of lateritic units with significant thickness variations, but in general they appear as shown in Figure 7-5.

Figure 7-5 Typical weathering profile comparing HZM units with Vale's logging codes



Source: HZM, 2019

The mineralisation at Vermelho is contained within intensely weathered serpentinites, weathered to saprolites and limonites. These weathering products are characterised by extensive silicification, generally in the form of veins, boxworks and massive zones of chalcedonic silica (from top to bottom):

- The soil (coded COB) is the overburden, formed of fine ferruginous soil, organic residues, and ferruginous concretions, commonly with rounded shapes (1 m average thickness). It represents the extreme leaching of the weathering profile, with relative enrichment of Fe_2O_3 and Al_2O_3. Usually under the ferruginous carapace (ferricrete) is two or three metres of ferruginous limonite (coded SAPFE), without ferruginous concretions.
- The silica horizon, coded SILICA, formed of thick, very resistant massive silica crusts, with SiO_2 content usually above 65%, sometimes reaching 90%, this also represents intense leaching, but not as intense as that of soil or ferruginous limonite (30 m average thickness). It is this chalcedony carapace that supports the topography, preserving the deposit from erosion. It develops exclusively on dunitic and peridotitic rocks. Features irregular and discontinuous siliceous limonite (SAPSIL) and ferruginous limonite (SAPFE) pockets. The nickel grades rarely exceed 0.25%.
- Underneath the SILICA is the siliceous limonite layer (SAPSIL), with varying proportions between oxides and hydroxides of iron and silica with SiO_2 content ranging from 35% to 65%, and boxwork structures are usually packed with Fe oxides and hydroxides (20 m average thickness). Frequently in V1 there are thick and continuous levels of sub-horizontal ferruginous limonite within the siliceous limonite. The nickel grades range from 0.6% to 1.5%.

- The next layer, ferruginous limonite (SAPFE) is composed of finely grained and porous Fe oxides and hydroxides (goethite, limonite, hematite), and forming discontinuous and irregular lenses within the siliceous limonite or the massive silica; the Fe_2O_3 content commonly exceeds 80%, and the SiO_2 content is lower than 35%; the nickel grades range between 0.6% and 1.5% (15 m average thickness).
- The saprolite (SAP) layer composes a constant level overlapping weathering/fresh rock surface. In general, its top is strongly silicified (2–3 m). The contact between limonite and saprolite can be abrupt or transitional. In the Vermelho drillhole database, it is characterised by the predominance of serpentinite and relict structures, and MgO above 10%; the nickel grades usually exceed 1.5%, sometimes reaching 10% (15 m average thickness).

Contacts between these zones are commonly transitional. Other materials can be present in different proportions and positions in the lateritic profile: weathered and semi-weathered serpentinites (coded SERPINT and SERPSI, respectively), fresh serpentinites (coded SERP), pyroxenitic and gabbroic saprolites (coded SAPIROX and SAPGAB, respectively), and fresh gabbros and pyroxenites (coded GAB and PIROX, respectively).

7.6 Mineralisation

At Vermelho, two main mineralisation types are observed: silicate at the base (saprolite) and oxide at the top (limonite) of the weathering profile.

- Limonite: the oxide material, with 1.2% nickel oxide (NiO) on average, is composed predominantly of goethite, and also contains chlorite, spinels and silica. In this case, the nickel is highly concentrated in chlorite (average 12% NiO), whereas in goethite NiO content ranges between 0.9% and 1.7%. As a result, the presence of chlorite, even in minor quantities, is important in elevating the grade of the oxide ore. Locally, higher grades of mineralisation can also be due to the presence of nickeliferous smectites.
- Saprolite: the mineralogical composition of the silicate zone, with 1.8% NiO in average, consists largely of serpentine, chlorite, and spinels, with quartz and goethite in minor amounts. Serpentine and chlorite are the main nickel-bearing minerals, nickel being about equally distributed between the two phases (2% to 3% NiO). There is no significant development of an enriched transition mineralisation type between the oxide and silicate horizons.

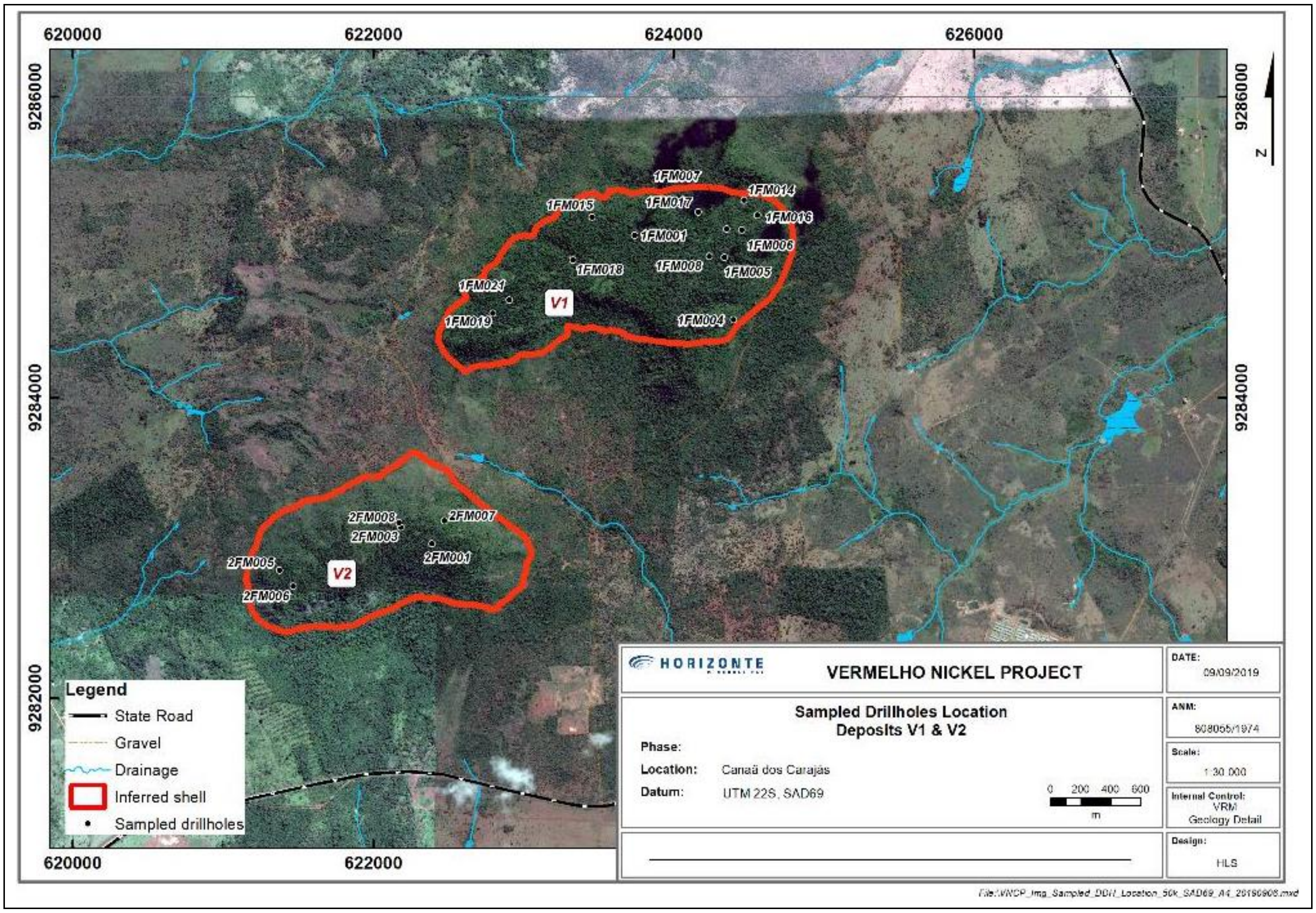
Barren zones within the lateritic profile are represented by serpentinite blocks, strongly silicified bands, mafic or pyroxenitic dykes, and ferruginous concretions (common at the top of the section). A transitional saprolitic zone occurs at the base of the mineralised zone, with nickel grades usually ranging from 0.25% and 0.5%.

The Vermelho laterites are mineralogically similar to the limonitic laterites of Western Australia and, like the Australian ores, are amenable to pressure acid leaching. A possible treatment route of the higher grade saprolite ores is the RKEF process as envisaged for HZM's Araguaia deposits.

7.7 Mineralogy

Vale carried out mineralogical studies in 2003 and 2004 in order to understand the variations among the mineralised horizons to use in the development of the proposed beneficiation process. Thus, 50 samples previously selected for leaching tests were characterised. This selection was made based on the representativeness of the lithotypes in the deposit and the variation of nickel content. Each of these samples was evaluated in the –200# leaching feed fractions and +200# beneficiation rejects (Figure 7-6).

Figure 7-6 Sampled holes location



Source: HZM, 2019

Within the mineralised zone, three principal units can be recognised, two in limonite, named ferruginous (SAPFE) and siliceous (SAPSIL) limonite, and one in saprolite (SAP) (Figure 7-7). These three units demonstrated unique behaviours in testing, due to their mineralogical content and texture, reflecting the geological evolution of the deposit.

Figure 7-7 Mineralised units

Notes:

- *Siliceous limonite sample.*
- *B – Ferruginous limonite sample.*
- *C – Saprolite sample.*

Source: HZM, 2019

Siliceous limonite (SAPSIL) is characterised by the abundance of quartz/chalcedony generally in a boxwork texture (Figure 7-7 A), some of which persists in the leach feed samples (10% of the total mass).

In the samples where quartz/chalcedony is more abundant, it was found that it is fine grained, with crystals sizes of less than 50 μm. Nevertheless, the beneficiation process proved very effective at removal of this mineral (the quartz/chalcedony concentration in the tailings is 60% to 90% (Table 7-1)).

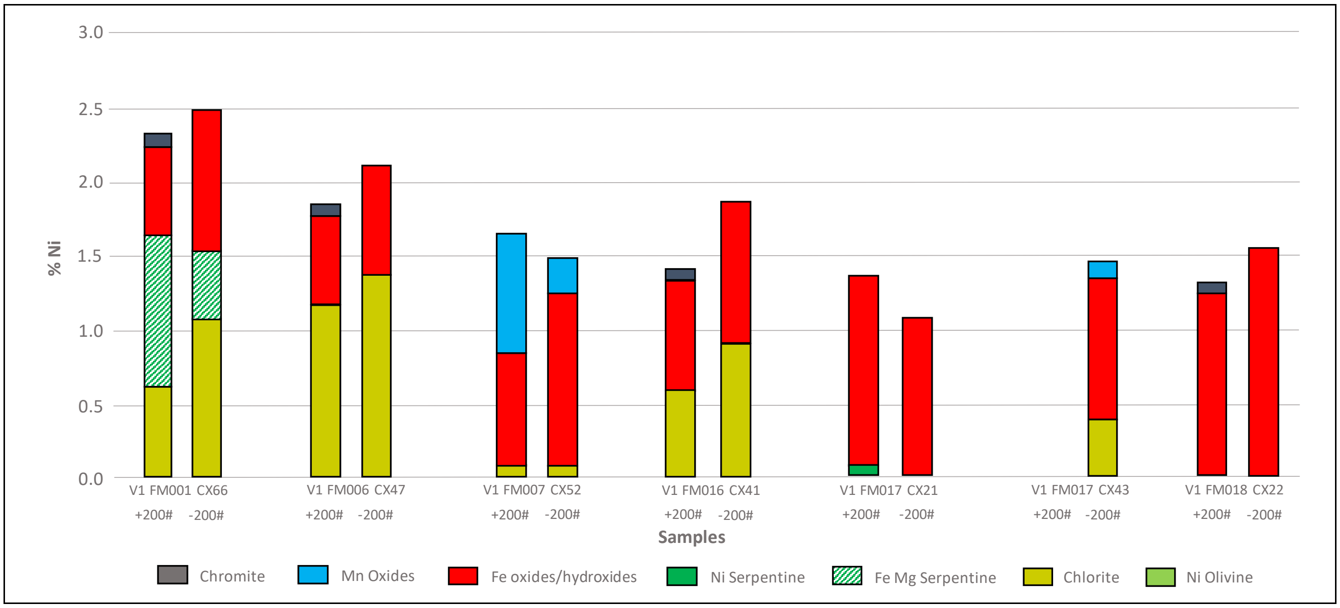
Table 7-1 Average percentage of minerals in siliceous limonite

Minerals	Formula	-200# (Leach feed)		+200# (Rejects)	
		Average	Standard deviation	Average	Standard deviation
Quartz/Chalcedony	SiO_2	10	8	60	24
Kaolinite	$Al_2Si_2O_5(OH)_4$	1	1	t	0,3
Chlorite	$(Fe,Mg,Fe)_5Al(Si_3Al)O_{10}(OH,O)_8$	5	10	5	11
Fe-Mg Serpentine	$(Mg,Fe)_3Si_2O_5(OH4)$	9	20	2	4
Ni Serpentine	$(Mg,Ni)_2Si_2O_5(OH)_2$	<1	2	t	1
Olivine Ni	$(Ni,Mg)_2SiO_4$	rr	0,1	-	-
Anatase/Rutile	TiO_2	rr	0,1	-	-
Gibbsite	$Al(OH)_3$	-	-	r	0,4
Brucite	Mg(OH)	1	4	-	-
Carbonate	$MgCO_3$	rr	-	-	-
Goethite/Agreg. limonitic	FeO.αOH	46	20	13	16
Hematite/Martite	Fe_2O_3	26	11	13	11
Magnetite	Fe_3O_4	rr	0,1	-	-
Mn oxides	$(Co,Ni)_{1-y}(MnO_2)_2\text{-}x(OH)_{2-2y+2x}\cdot n(H_2O)$	<1	1	t	1
Chromite	$FeCr_2O_4$	1	1	6	6
Ilmenite	$FeTiO_3$	rr	-	-	-

Note: "-" Not detected; rr (very rare): ~0.05%; r (rare): ~0.2%; t (trace): ~0.5%; <1: ~0.8%.

The average percentage of chlorite and Fe-Mg serpentine in –200# samples are 5% and 10%, respectively, and 5% and 2% in the +200# rejects. Some of the chlorites present in the +200# fraction of siliceous limonite are coarse and the others are included in quartz/chalcedony particles, which eventually justifies their presence in the leach feed (Figure 7-8).

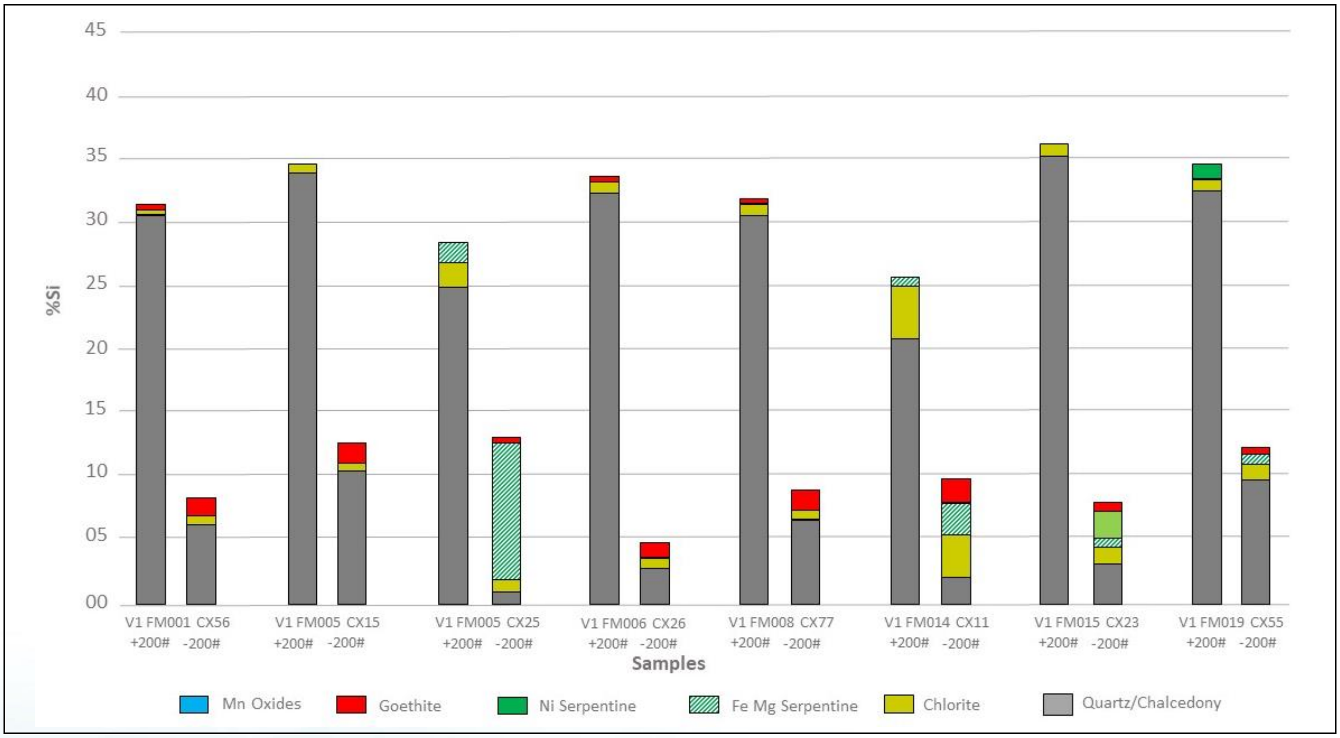
Figure 7-8 Nickel deportment in siliceous limonite



Source: Vale, 2004

The beneficiation of the siliceous limonite produced a very rich quartz tailing, which accounts for almost all the silica contained in some samples of the +200# fraction. In contrast, in the –200# fraction, the occurrence of silica in the form of quartz is reduced by increased silica from other silicates such as chlorite and serpentine, as well as from iron hydroxides. Nevertheless, quartz remains one of the main sources of silica in the siliceous limonite leach feed and its persistence in the fine fraction is likely related to textural aspects such as crystal size (Figure 7-9).

Figure 7-9 Silica deportment in siliceous limonite



Source: Vale, 2004

In the ferruginous limonite (SAPFE) unit (Figure 7-7 B), goethite/limonite and hematite, with concentrations ranging from 41% to 75% and from 17% to 46% respectively, dominate in the –200# fraction. Due to the abundance of these minerals, coupled with the low silicate content, at least in the most typical samples, the compositional variations between leaching and beneficiation rejects would be less significant. This means that the mineralogical composition is practically homogeneous, and the iron oxides/hydroxides would be differentiated by density.

Iron hydroxides tend to concentrate slightly in the finer fraction, averaging 56% versus 48% on the +200# fraction. Hematite, on the other hand, is almost evenly distributed between both fractions (mean 33%), as it can occur as both fine and coarse particles (Table 7-2).

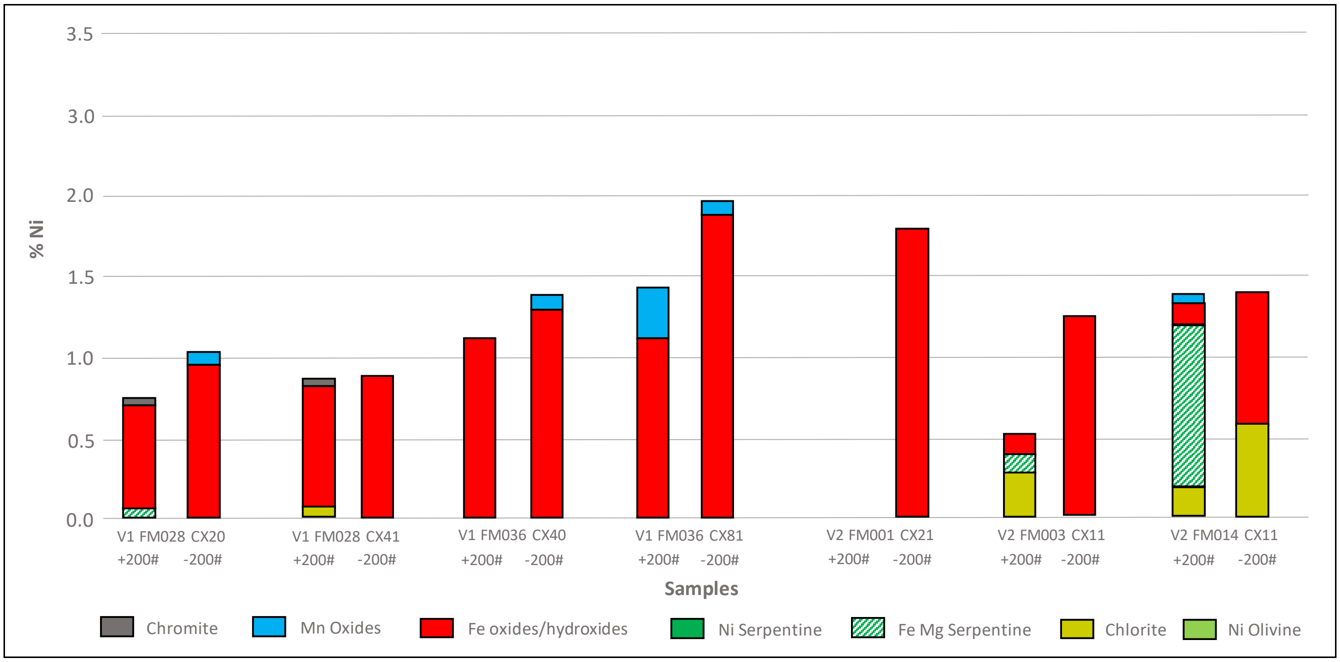
Table 7-2 Average percentage of minerals in ferruginous limonite

Minerals	Formula	-200# (Leach feed)		+200# (Rejects)	
		Average	Standard deviation	Average	Standard deviation
Quartz/Chalcedony	SiO_2	2	2	4	22
Kaolinite	$Al_2Si_2O_5(OH)_4$	2	3	<1	1
Chlorite	$(Fe,Mg,Fe)_5Al(Si_3Al)O_{10}(OH,O)_8$	3	4	2	3
Fe-Mg Serpentine	$(Mg,Fe)_3Si_2O_5(OH_4)$	1	5	4	12
Ni Serpentine	$(Mg,Ni)_2Si_2O_5(OH)_2$	-	-	rr	-
Olivine Ni	$(Ni,Mg)_2SiO_4$	-	-	-	-
Anatase/Rutile	TiO_2	-	-	-	-
Gibbsite	$Al(OH)_3$	-	-	r	0,5
Brucite	$Mg(OH)$	-	-	-	-
Carbonate	$MgCO_3$	-	-	-	-
Goethite/Agreg. limonitic	$FeO.\alpha OH$	56	11	48	17
Hematite/Martite	Fe_2O_3	34	10	33	13
Magnetite	Fe_3O_4	r	0,3	-	-
Mn oxides	$(Co,Ni)_{1-y}(MnO_2)_2\text{-}x(OH)_{2-2y+2x}\cdot n(H_2O)$	<1	1	2	4
Chromite	$FeCr_2O_4$	1	1	6	3
Ilmenite	$FeTiO_3$	-	-	-	-

Note: "-" Not detected; rr (very rare): ~0.05%; r (rare): ~0.2%; t (trace): ~0.5% ; <1: ~0.8%.

Despite containing at most 3.0% Ni, iron oxides/hydroxides are responsible for most of the nickel present in ferruginous limonite samples (Figure 7-10). The low degree of enrichment observed during the beneficiation stages after elimination of the coarser fraction may be caused by the either the absence or low concentrations of quartz, resulting in a uniform mineralogical composition in the –200# and +200# fractions, except for the evident chromite. There is practically no concentration of nickel-bearing minerals. The mass split between these two fractions, therefore, would be conditional on the existence of goethite occurring as fine particles, compact coarse particles or even as fine agglomerates.

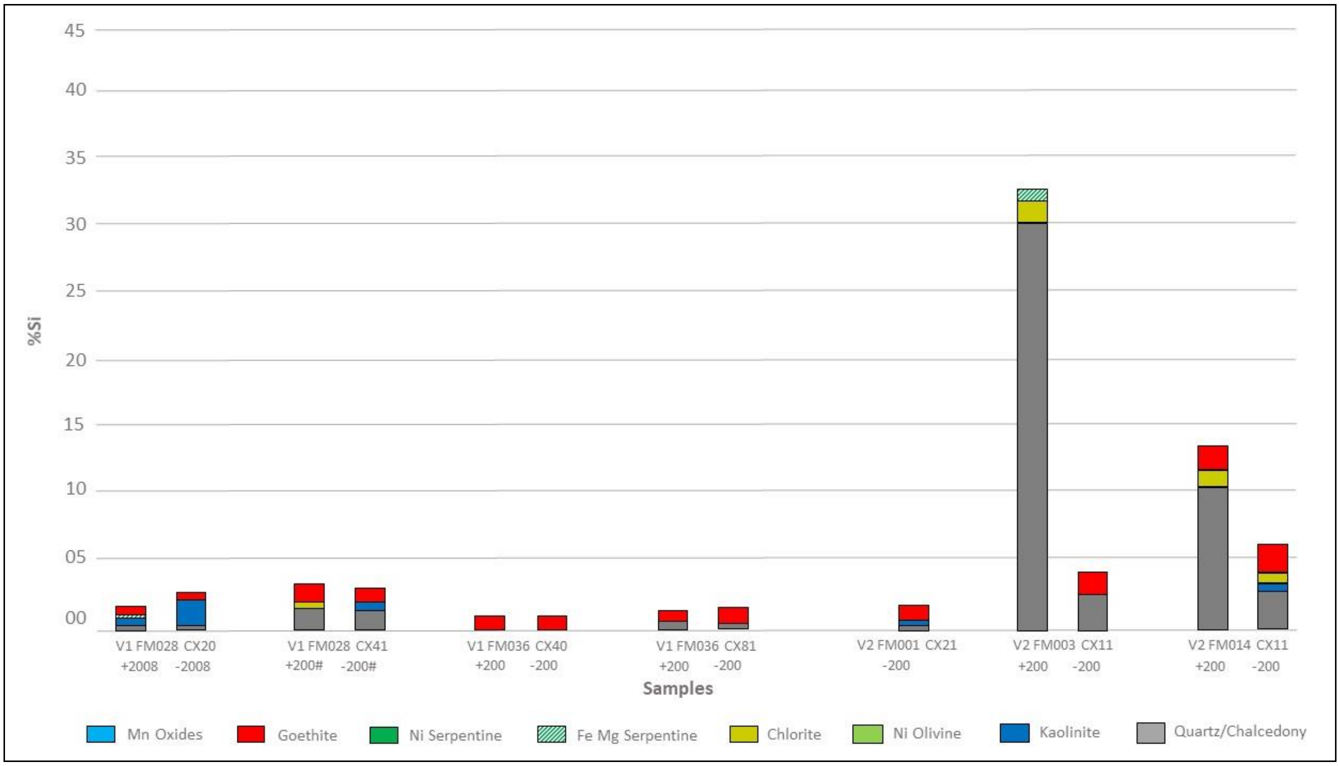
Figure 7-10 Nickel deportment in ferruginous limonite



Source: Vale, 2004

In ferruginous limonite, some silica is from quartz and some from iron hydroxides. The existence of silica in these phases is the result of the partial transformation from the serpentine, and its persistence in the crystalline structure prevents the formation of pure goethite, which result in a denser structure. This distribution supports the observation that the mass split between the –200# and +200# fractions does not necessarily indicate mineralogical segregation, since the composition and hence the origin of the elements is the same. Some samples are composed of high kaolinite contents, and that also influences the distribution of silica (Figure 7-11).

Figure 7-11 Silica deportment in ferruginous limonite



Source: Vale, 2004

Saprolite (SAP) consists of samples with a high percentage of serpentine, which on average represents 32% of the total mass in the –200# fraction and may reach more than 75% (Figure 7-7 C). The high concentration of this mineral, however, does not guarantee that the nickel content is equally high, as this element is less concentrated compared to the same minerals in the siliceous limonite or even ferruginous limonite samples (Table 7-3).

The beneficiation was not as efficient in the separation of quartz and Mg silicates (serpentine and chlorite) as in siliceous limonite. In the +200# fraction of saprolite, although the quartz content is slightly higher, the serpentine percentage may reach over 50% in some cases, and chlorite appears on average to be 10%, which means considerable loss of minerals carrying nickel.

In mineralogical terms, the existence of Mg silicates in the coarse fraction can be justified by textural factors such as high particle size and smaller porosity which, ultimately, are directly related to the greater preservation of this type of ore under weathering conditions.

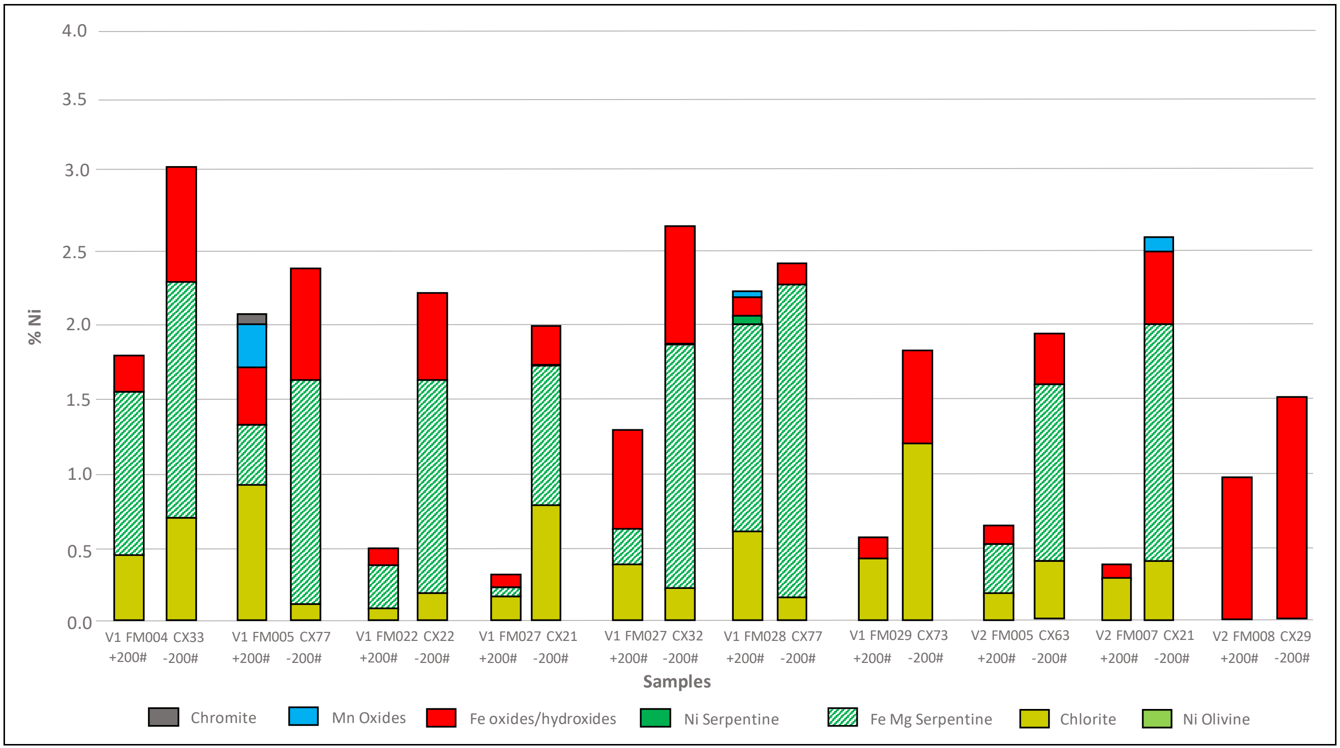
Table 7-3 Average percentage of minerals in saprolite

Minerals	Formula	-200# (leach feed)		+200# (rejects)	
		Average	Standard deviation	Average	Standard deviation
Quartz/Chalcedony	SiO_2	6	5	43	33
Kaolinite	$Al_2Si_2O_5(OH)_4$	r	0,3	r	0,2
Chlorite	$(Fe,Mg,Fe)_5Al(Si_3Al)O_{10}(OH,O)_8$	6	4	9	8
Fe-Mg Serpentine	$(Mg,Fe)_3Si_2O_5(OH4)$	32	24	13	18
Ni Serpentine	$(Mg,Ni)_2Si_2O_5(OH)_2$	-	-	r	-
Olivine Ni	$(Ni,Mg)_2SiO_4$	-	-	-	-
Anatase/Rutile	TiO_2	rr	0,1	-	-
Gibbsite	$Al(OH)_3$	-	-	-	-
Brucite	Mg(OH)	-	-	-	-
Carbonate	$MgCO_3$	-	-	-	-
Goethite/Agreg. limonitic	$FeO.\alpha OH$	31	20	13	17
Hematite/ Martite	Fe_2O_3	23	13	14	9
Magnetite	Fe_3O_4	rr	0,1	-	-
Mn oxides	$(Co,Ni)_{1-y}(MnO_2)_2\text{-}x(OH)_{2-2y+2x}\cdot n(H_2O)$	t	0,3	1	2
Chromite	$FeCr_2O_4$	1	1	7	4
Ilmenite	$FeTiO_3$	-	-	-	-

Note: "-" Not detected; rr (very rare): ~0.05%; r (rare): ~0.2%; t (trace): ~0.5%; <1: ~0.8%.

At Vermelho, the saprolite is less affected by weathering than is typical. Mg silicates would have been subjected to only moderate concentration of nickel, with incipient silica and Mg leaching and there would have been no enrichment caused by nickel precipitation from ferruginous limonite descending solutions (Figure 7-12). As a result, the concentrations are lower but offset by the abundance of other minerals. The comparison between nickel levels in Mg silicates from different lithotypes is more evident for chlorite: in siliceous limonite, nickel levels are generally higher than 10% whereas the average for ferruginous limonite is around 9%. In saprolite, it is much lower, around 6%. This is consistent with the genetic evolution of lateritic nickel deposits.

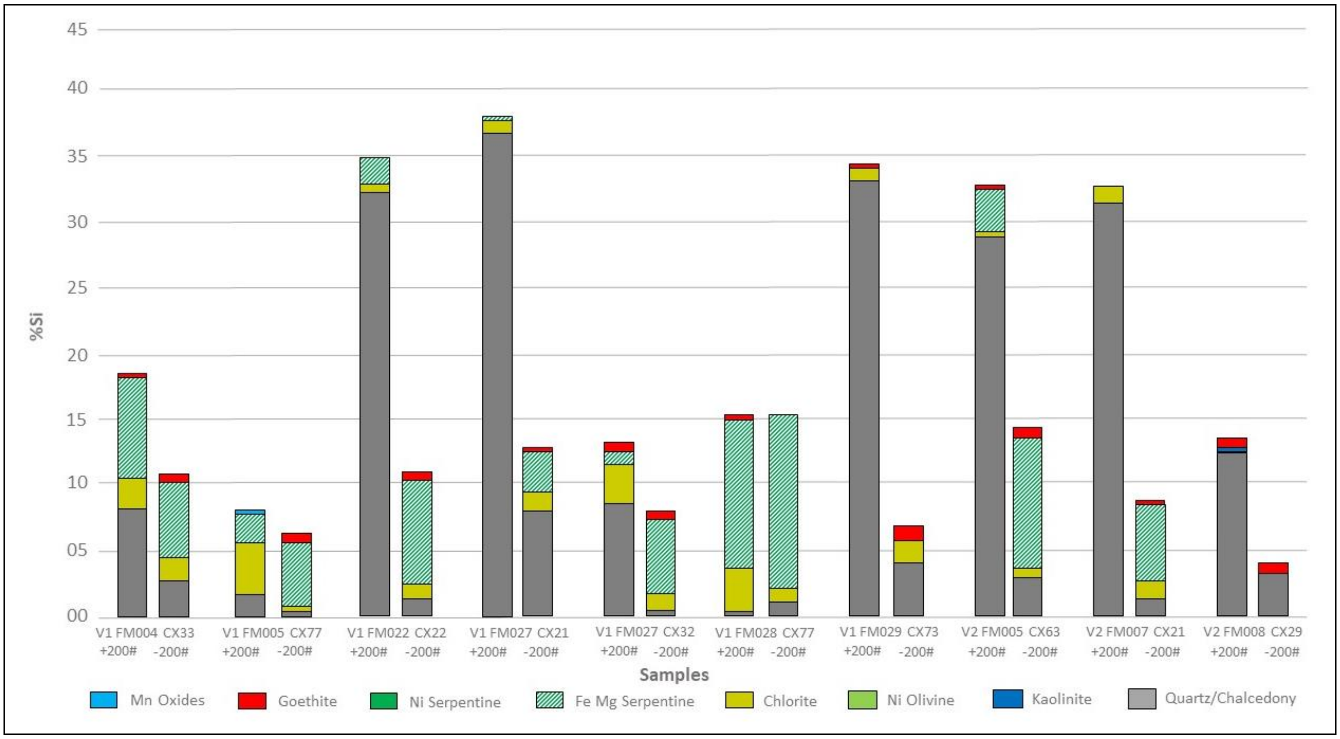
Figure 7-12 Nickel deportment in saprolite



Source: Vale, 2004

For saprolite samples with characteristics similar to siliceous limonite, efficient quartz removal is also observed. This explains why saprolite lithotypes returned positive beneficiation test results, albeit not as compelling as the siliceous limonite (Section 13.2.3). However, for typical samples, a large portion of the silica contained in the tailings originates from the Mg (serpentine and chlorite) silicates themselves, which would indirectly indicate the significant loss of nickel itself. One of the samples that best exemplifies this fact is V1 FM004 cx33 +200#, with more than 8% silica from quartz and approximately 10% from Mg serpentine and chlorite silicates (Figure 7-13).

Figure 7-13 Silica deportment in saprolite



Source: Vale, 2004

The beneficiation process was comparatively more efficient in removing quartz from siliceous limonite. In saprolite, the separation of quartz and Mg silicates seems to be more complicated, since both mineral species are detected in the two particle size fractions, causing loss of nickel in the rejects and increase of silica in the leach feed. This would be related to textural aspects, such as large particle size, quartz inclusion and low porosity, due to the poor weathering of saprolite. In addition, nickel appears more dispersed, as it occurs at lower levels in Mg silicates of this unit. The incipient enrichment of ferruginous limonite in the leach feed would be due to the mineralogical homogeneity, or the low percentage of mineral species other than Fe oxides/hydroxides.

7.8 Geological interpretation and modelling

For geological modelling, Vale used a combination of wireframe solids and surfaces based on drillhole logging and assays. For solid modelling, the classical method of manual interpretation of the lithological contacts by sections was undertaken; the method considers an orthogonal projection to the section plan to include drillhole information that does not sit exactly in the section plan. This projection is halfway to the previous and next section. For surface modelling, interpolated grids with cells of 12.5 m x 12.5 m were generated based on existing drillhole intersections.

At Vermelho, vertical sections are spaced every 25 m with an east-west direction. Horizontal sections, also used for interpretation, are distributed every 10 m.

At total of 1,517 holes were used for V1 modelling, and 1,022 holes at V2.

Using both sets of sections, the model units were clustered into the following groups: geologic, weathering, chemical, and ore.

For surface modelling, criteria were developed to pick up the drillhole intersections that would define the limits. Table 7-4 presents the criteria used for modelling each surface.

Table 7-4 Surface modelling criteria

Surface	Criteria
COB	Cover description starting at the bore opening; COB definitions below rock were not considered, below or coinciding with a topographic surface
Al_2O_3	First peak with Al_2O_3 >2%, below or coinciding with the cover zone
MgO	Zone with continuous values, greater than 10% of MgO, below or coinciding with the Al_2O_3 limit
NiO^5	Zone with nickel grade greater than 0.5%, located below or coinciding with the MgO limit
NiO^{25}	Zone with nickel grade greater than 0.25%, located below or coinciding with the NiO^5 limit
Bedrock	Limited starting with the Serpsi descriptions, where it did not contradict the bore description, Serp, Gab and Pirox below or coinciding with the MgO limit

Source: AMEC, 2006

Where intersections in between surfaces occurred, an automated method of generating a minimum in between them was used. This method identifies crossings and fixes the lower surface to match with the upper one. After surfaces were created and fixed, solids representing each unit were created by joining top and bottom surfaces.

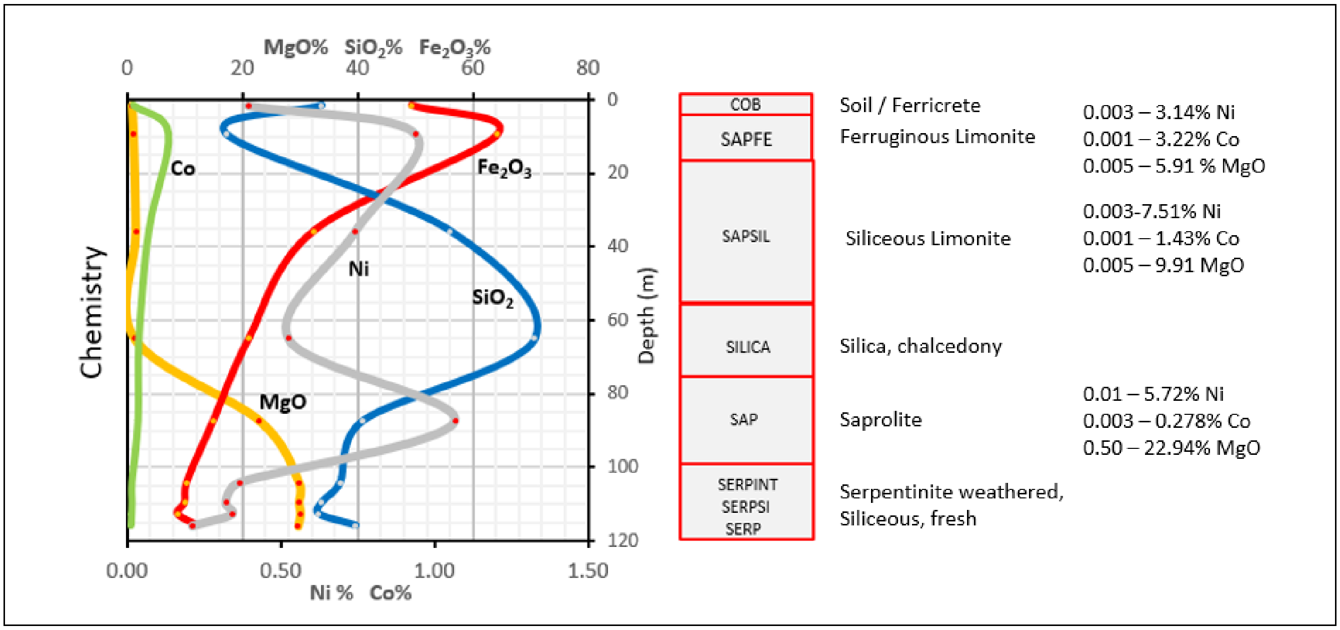
AMEC considered the methodology used of good practice and reviewed several sections for V1 and V2, found no problems. Only the ferruginous saprolite solids that were interpreted manually should have a more consistent extrapolation shaping.

8 DEPOSIT TYPES

The target mineralisation at the Project's V1 and V2 deposits 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 (pH – activity of hydrogen ions; Eh – activity of electrons).
- 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 100 m.
- 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 recrystallisation 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 two distinct horizons, limonite (oxide) and saprolite (silicate). The transition between these two horizons can be thick; however, in the Vermelho case the transition horizon is thin.

Figure 8-1 Chemical trends in schematic V2 nickel laterite profile



Source: HZM, 2019

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 manganese-cobalt-nickel 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 (greater than 0.5% Ni) and target the most prospective parts of a weathered ultramafic sequence.
- Follow-up drilling to delineate nickel-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 samples (Ni, Co, Mn, Cr, Mg, Fe, Si, Al and ignition loss).

9 EXPLORATION

HZM has not conducted any exploration work on the Vermelho licence as all exploration work on the licence was completed prior to HZM announcing the acquisition of the Project in December 2017.

Prior exploration work in the region has been conducted by DOCEGEO (a subsidiary of Vale) and contractors.

The following is extracted from AMEC (2006), reviewed and amended by the author for inclusion in this Technical Report.

9.1 Surveys

L&I Geologia e Topografia Ltda (later named Geotec Geologia e Topografia Ltda) was the local contractor in charge of the topographical services. A local coordinate system was initially used for referencing, but in 2002 the local system was converted to UTM. The first topographic map was at 1:10,000 scale, with contour lines every 10 m (locally every 5 m). The SAD69 datum was used.

A regular grid with 275° azimuth was surveyed at V1, and later continued into V2 with the same orientation. In V1, Vale cut 4.5 km base lines and control polygonal lines, as well as 70 km of perpendicular lines placed 50 m apart from each other. In V2, there were 4 km base lines and control polygonal lines, as well as 76 km of perpendicular lines placed 50 m apart from each other. The lines had wooden pickets every 20 m, and cement monuments every 400 m.

The whole survey was conducted with total station equipment. The polygonal accuracy is as follows:
$\Delta x \leq 0.3$ m; $\Delta y \leq 0.3$ m; and $\Delta z \leq 0.8$ m.

Exploration has been mainly by drilling and pitting. Drilling is described in Section 10.

10 DRILLING

HZM has not conducted any drilling work on the Vermelho licence as all drilling work on the licence was completed prior to HZM announcing the acquisition of the Project in December 2017.

The following is extracted from Snowden (2005), AMEC (2006) and GRD-Minproc (2005), reviewed and amended by the author for inclusion in this Technical Report.

10.1 Drilling and pitting summary

Vale has conducted both core and RC drilling at V1 and V2 since 1974. However, no details were provided to AMEC for the drilling methodologies utilised before 2002.

The Vale PFS resource estimate was based largely upon a 50 m x 50 m square drilling pattern. Infill drilling was undertaken to a 25 m x 25 m grid over much of V2 and around the edges of V1. This drilling was largely designed to convert areas that had been classified as Inferred in the PFS to Measured or Indicated. Four small areas (three from V2 and one from V1) of close spaced drilling (to a 12.5 m x 12.5 m pattern) was undertaken to provide detailed geological information and to determine the short-range variability of the grade components for resource modelling.

RC drilling between 2002 and 2004 at the V1 and V2 deposits was performed by Geosedna, on 100 m x 100 m and 50 m x 50 m grids (sometimes with a hole in the centre of the 50 m x 50 m cell), and locally on a 12.5 m x 12.5 m grid. Geosedna used three Prospector W750 rigs, with 4-¼” and 5-1/8” (108 mm and 130 mm respectively) diameters (Table 10-1). The maximum hole lengths were 175 m in V1 and 115 m in V2, and the average hole lengths were 56.4 m in V1 and 58.0 m in V2. The average recovery at both areas was 80%.

Table 10-1 Drillholes by deposit

Deposit	Year	Type	No. of holes	Meterage (m)	Average (m)	Diameter	Grid
V1	1976–1998	RC	15	1,172	-	5-1/8”	Partial data
		DDH	4	305	-	HQ	Partial data
	2000–2001	RC	113	7,490	-	5-1/8”	100 m x 100 m
		DDH	2	160	-	HQ	-
	2002–2003	RC	490	31,356	-	4-¼”	50 m x 50 m
		DDH	38	1,961	-	ZW	-
	2003–2004	RC	651	31,553	-	4-¼”	50 m x 50 m; 12.5 m x 12.5 m
		DDH	22	1,956	-	HQ	-
		DDH	48	1,622	-	ZW	-
	Subtotal		**1,383**	**77,575**	**56.1**		
V2	2002–2003	RC	229	15,265	-	4-¼”	100 m x 100 m
		DDH	16	786	-	ZW	-
	2003–2004	RC	620	34,034	-	4-¼”	50 m x 50 m
		DDH	12	1,080	-	HQ	-
	Subtotal		**877**	**51,165**	**58.3**	**-**	**-**
Total			**2,260**	**128,740**	**57.0**		

Notes: 4-1/4” = 108 mm; 5-1/8” = 130 mm; HQ = 63.5 mm; ZW = 202.5 mm; RC = reverse circulation; DDH = diamond drilling.

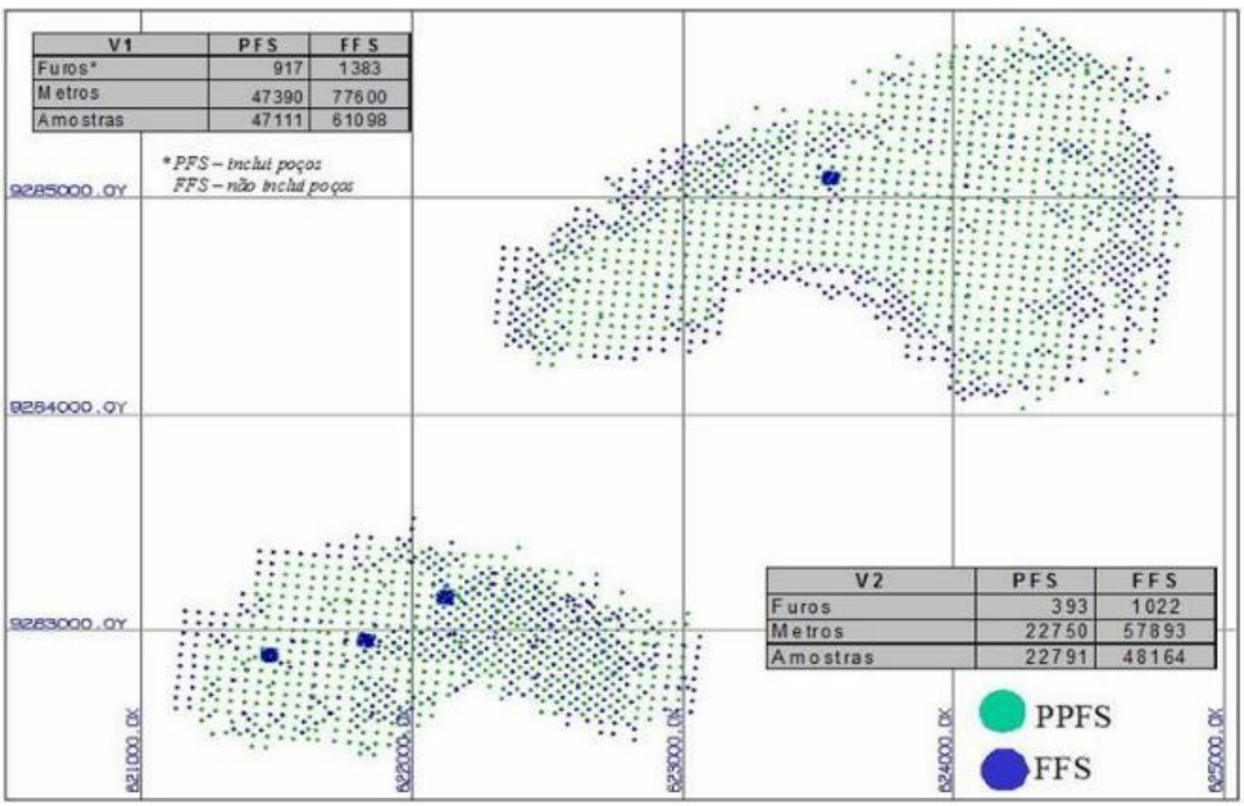
Source: AMEC, 2006

Vale also conducted diamond core drilling in the Vermelho deposits. Geosol, a local contractor, used three Maqsonda rigs to drill 40 drillholes with HQ (or 96.1 mm) diameter and at variable inclinations from 60° to 90° (Table 10-1). The vertical diamond holes always twinned previously drilled RC holes, and the inclined holes were oriented to intercept shear zones or dykes. Maximum hole lengths were 146.0 m in V1 and 115.6 m in V2, and the average hole lengths were 86.5 m in V1 and 90.0 m in V2. Average core recoveries were 91% and 95%, respectively.

Geosol drilled 102 large diameter holes (ZW, or 150 mm) with Maqsonda rigs (Table 10-1). These holes also twinned RC holes, and the recovered material was used for mineralogical and metallurgical studies. The maximum hole lengths were 85.0 m in V1 and 93.1 m in V2, and the average hole lengths were 41.7 m in V1 and 48.3 m in V2.

All holes were marked with large wooden posts and aluminium labels, including the drillhole number and the UTM coordinates, and some holes were also marked with cement blocks with similar identifications. Figure 10-1 shows the distribution of the drillhole grids.

Figure 10-1 Location of drillholes and pits for the PFS and FFS



Note: Holes (Furos); Metres (Metros); Samples (Amostras)

Source: GRD-Minproc, 2005

Shallow pits were dug to provide geological information in two campaigns, an early program from 1996 to 1998 that was restricted to V1, and a later program, in 2003 to 2004 that covered both deposits. Pitting details are shown in Table 10-2.

Table 10-2 Pitting by deposit

Deposit	Year	No. of pits	Meterage (m)	Dimensions
V1	1976–1998	114	1,792	1 m x 1 m
	2003–2004	20	318	1 m x 1 m
	Subtotal	**134**	**2,111**	
V2	2003–2004	9	6,548	1 m x 1 m
	Subtotal	**9**	**6,548**	
Total		**143**	**8,659**	

Source: AMEC, 2006

10.2 Logging and sampling

The RC samples were taken at 1 m intervals by collecting the entire discharge from the bottom of the cyclone. This material was immediately weighed and bagged in thick plastic bags. The samples were subsequently placed on aluminium trays, dried on a charcoal oven, and re-bagged. Drilling weight recovery was estimated by comparing the actual sample weight with the theoretical sample weight based on the interval drilled, the hole size and the material dry density. The samples were later quartered, initially through the cone-and-quartering method, and later using a Jones splitter, to obtain 2–3 kg samples which were again split in two portions: one to be submitted to the preparation facility (for pulverisation) and one to be kept as for future reference.

From the reference samples, a small portion was separated and placed on a description board (one per drillhole), consisting of 1.5 m x 0.1 m rectangular pieces of wood, where the cuttings were glued forming 2 cm-wide bands, one per 1 m drilled (Figure 10-2). These description tables served for future fast reference regarding lithology, etc. A second portion of the backup sample was stored in wood boxes divided in cells (Figure 10-3).

Figure 10-2 Description board

Source: AMEC, 2006

Figure 10-3 RC cuttings stored in wood boxes

Source: AMEC, 2006

Sample lithology was described using the RC cuttings. In total, the project geologists differentiated 13 lithologies, which are listed in Table 10-3 (including the corresponding codes). The HZM technical team elected to update the nomenclature of the weathering units so that they are in alignment with HZM's other projects (refer to Section 7.4 for details).

Table 10-3 Lithological codes used at Vermelho

Lithology (Vale)	Code	Units (HZM)
Cover	COB	Soil/Silcrete
Ferruginous saprolite	SAPFE	Ferruginous limonite
Siliceous saprolite	SAPSIL	Siliceous limonite
Silica	SILICA	Silica
Saprolite	SAP	Saprolite
Weathered serpentinite	SERPINT	Weathered serpentinite
Semi-weathered serpentinite	SERPSI	Semi-weathered serpentinite
Fresh serpentinite	SERP	Fresh serpentinite
Pyroxenite saprolite	SAPIROX	Pyroxenite saprolite
Pyroxenite	PIROX	Pyroxenite
Gabbroic saprolite	SAPGAB	Gabbroic saprolite
Gabbro	GAB	Gabbro
Granite	GRANITO	Granite

Source: AMEC, 2006

Diamond drill core was stored in wood boxes and transported to a facility at the camp, where they were logged and sampled. The logs were general and mainly described the rock type (as per Table 10-3) and colour, the presence of some characteristic minerals (such as garnierite, chromite, chlorite, plagioclase, manganese oxides) and in few cases, certain structural features. The distinction between mineralised and waste units was quite clear in drill core. All these attributes were hand-entered into the Project database management system.

The choice of lithological units and the methods for core logging are suitable to separate mineralised and non-mineralised units and are appropriate for this type of deposit.

After logging, the core was sampled as quarter-core samples, in some cases with a mechanical splitter and in others with a diamond saw. Sample intervals were marked off in nominal 1 m intervals. However, main geological contacts were honoured; maximum and minimum sample lengths were 2.6 m and 0.3 m, respectively.

The large diameter core was logged, photographed, and finally sampled with 3 cm wide x 3 cm deep x 1 m long channels taken along the core axis. The magnetic susceptibility was determined in the sample bags with a handheld Exploranum kappameter.

A summary of sampling at V1 and V2 is presented in Table 10-4.

Table 10-4 V1 and V2 sampling summary

Deposit	No. of samples	Length				Length in mineralisation			
		Total (m)	Maximum (m)	Minimum (m)	Average (m)	Total (m)	Maximum (m)	Minimum (m)	Average (m)
V1	78,718	79,686	2.5	0.3	1.0	58,245	2.6	0.3	1.0
V2	57,698	57,561	1.6	0.4	1.0	43,530	1.6	0.4	1.0
Total	**136,416**	**137,246**	**-**	**-**	**-**	**101,776**	**-**	**-**	**-**

Source: AMEC, 2006

The choice of sampling intervals and sampling methods are appropriate for this type of deposit (where the transition in key ore types may occur over short downhole distances).

10.3 Statistical analysis of sample data

AMEC verified the statistical analysis of available information, such as:

- Assays of samples of vertical channels in pits
- Assays of samples of diamond cores
- Assays of samples of RC of rotary percussion
- Assays of samples of large-diameter diamond cores
- Measurements of in situ density (a distinct drilling database).

Sample support is 1 m long at the several sources.

Due to the complexity of the geological model, units were grouped by ore type to facilitate the analysis. Figure 10-4 and Figure 10-5 show the grouping criteria for V1 and V2, respectively.

Pit channel samples presented high discrepancies compared to other sampling sources, and Vale decided to ignore those samples when estimating resources. AMEC did not see problems in discarding such data, since enough information was available to obtain a good quality estimate.

AMEC considered that unit grouping was properly defined and executed to adequately represent the ore type variations at the deposit and provided a good basis for using this information in resource estimation.

In general, low coefficients of variation were obtained, even considering the original 1 m sample lengths.

Figure 10-4 Unit grouping criteria for V1

Assay Table

	Codes Ore	Data Total	Data Analyzed
0	1103	44	44
0	2103	8	8
0	2202	59	59
0	2203	3.774	3.759
0	2206	517	515
0	2301	343	343
0	2302	2.796	2.795
0	2303	355	354
0	2306	82	81
0	3301	47	47
0	3302	1.743	1.742
0	3303	560	559
0	3305	90	89
0	3306	61	57
0	3311	1	1
0	3312	51	51
0	3313	315	315
0	3316	15	15
0	3321	10	10
0	3322	533	533
0	3323	5.506	5.498
0	3324	1	1
0	3326	14	14
0	4103	1	1
0	4203	94	94
0	4206	9	9
0	5203	13	13
0	5204	8	8
0	5304	25	25
2100	1111	36	36
2100	1112	426	426
2100	1113	2.056	2.055
2100	1114	2	2
2100	1115	7	7
2100	1116	15	15
2100	2112	4	4
2100	2113	194	193
2100	4113	13	13
2210	2211	38	38
2210	2212	270	270
2210	2213	24.416	24.339
2210	5213	461	461
2210	5214	48	48
2310	2311	69	69
2310	2312	2.478	2.477
2310	2313	10.317	10.294
2310	4312	1	1
2310	4313	61	61
2310	5313	13	13
2310	5314	49	49
2320	2321	48	48
2320	2322	830	830
2320	2323	2.928	2.923
2320	4323	1	1
2320	5324	10	10
4210	4212	92	92
4210	4213	14.104	14.068
6000	2216	650	647
6000	2316	217	213
6000	2326	33	33
6000	4216	60	60
	TOTAL	59.947	59.796

Waste:
- BedRock - Thou-ending codes 3 (3____);
- Waste - 0-ending codes 0 (__0_).

- Geology codes grouped;
- High aluminum surface precedes gabro dyke code
- Codes 4113 (SAPFE) in High Aluminum.

- Geology codes grouped;
- Codes 5213 and 5214 (rockblock) .

- Geology codes grouped;
- Codes 5313 and 5314 (rockblock) .

- Geology codes grouped;
- Codes 5313 and 5314 (rockblock) .

- Geology codes grouped

Assay Table

Groups	Data Total	Data Analyzed
0	4.987	4.978
2100	1.676	1.674
2210	762	761
2310	23.606	23.561
2320	14.710	14.679
4210	5.030	5.015
6000	4.028	4.027
TOTAL	54.799	54.695

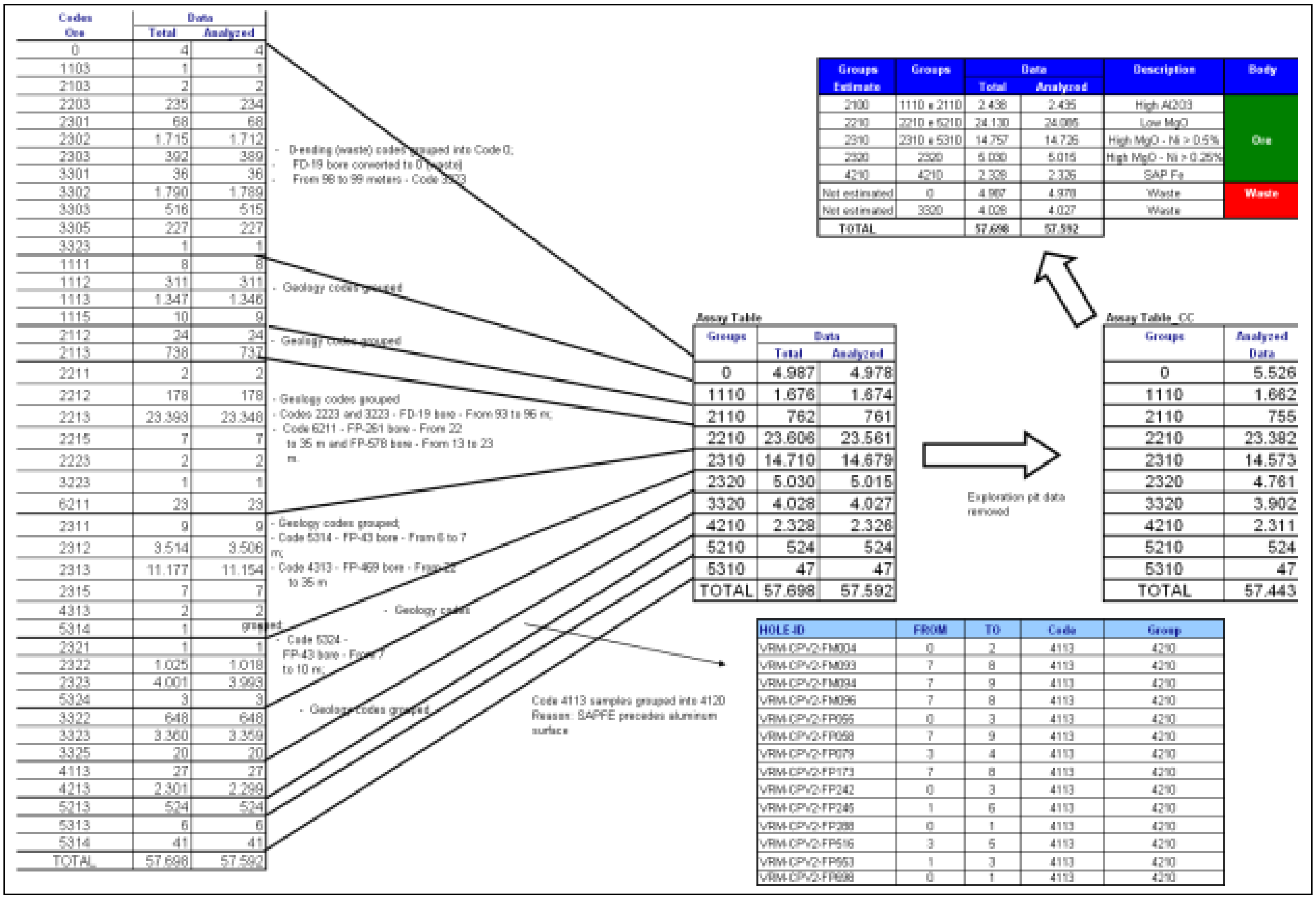
- Four-meter samples segmented in one meter;;
- Pit data removed:
  - Old pits: Location problems;
  - New pits: Difference greater than 20% in geminated data.

Assay Table _ CC

Group	Samples
2100	2753
2210	25233
2310	12988
2320	3817
4210	14196
6000	960
Ore Total	59947

Source: AMEC, 2006

Figure 10-5 Unit grouping criteria for V2



Source: AMEC, 2006

As described earlier, Vale drilled core holes, RC holes or excavated pits to twin RC holes and pits. This analysis was restricted for to the nickel variable only and to data whose horizontal distance was less than 5 m and 1 m vertical distance.

As can be observed in Table 10-5 and Table 10-6, V2 presents very high differences between Diamond Drill x Pit Channels and RC x RC samples. Such differences are not found in V1 in the same magnitude. At V1, the highest discrepancy is found when comparing Diamond Drill x Metallurgical samples and could be due to the sampling method used for the metallurgical holes, since a very small amount of material was sampled.

Table 10-5 Comparison of twin sample pairs of different sampling types for V1

Pair	Average			Pair distance			Pairs	Pair	
	1	2	Diff. (%)	Average	Minimum	Maximum	Counting	(An>0.4 x An<0.5)	(%)
FM x FM	0.93	0.93	-0.09	3.12	0.61	4.32	2,550	524	20.55
FM x FD	0.78	1.51	-63.87	45.89	45.89	45.89	48	9	18.75
FM x PO (new)	0.72	0.80	-10.57	2.09	0.87	3.08	44	6	13.64
FD x PO (old)	2.14	1.72	21.91	17.62	14.12	20.55	30	0	0.00
FP x PO (new)	0.68	0.86	-23.45	2.14	1.41	3.64	306	54	17.65
FP x PO (old)	1.08	1.21	-11.62	1.48	0.11	5.41	278	51	18.35
FP x FM	0.93	0.96	-2.77	1.90	0.00	5.80	3,110	729	23.44
FP x FD	0.77	0.83	-7.60	2.83	0.47	5.88	1,199	391	32.61
FP x FP	0.60	0.51	15.47	2.19	1.48	2.81	129	44	34.11

Notes: Metallurgical bores (FM); Rotary bores (FD), Exploration pits (PO) – disposed of; Roto-percussive bores (FP). 1 There are no pairs with distances. 2 Old pit data disposed of due to location problems and to the data themselves. New due to high grade difference (>20%) to. 3 AN>0.5 x AN<05 – Represents pair counting when one analysis was greater than and the other less than 0.5% Ni. The percentage column is the sample percentage under those conditions, which much be regarded as a project risk.

Source: AMEC, 2006

Table 10-6 Comparison of twin sample pairs of different sampling types for V2

Pair	Average			Pair distance			Pairs	Pair	
	1	2	Diff. (%)	Average	Minimum	Maximum	Counting	(An>0.4 x An<0.5)	(%)
FM x FM	1.07	1.08	-1.15	3.03	1.73	4.50	9,248	2,571	27.80
FM x FD	1.31	1.41	-5.99	3.40	1.04	4.75	310	99	31.94
FD x PO	0.73	2.00	-93.17	2.07	2.05	2.11	19	3	15.79
FP x PO	0.70	0.99	-34.70	2.89	0.73	5.80	148	49	33.11
FP x FM	1.19	1.08	10.01	2.08	0.21	5.60	6,412	1,725	26.90
FP x FD	0.98	0.95	3.09	2.62	0.44	5.54	647	236	36.48
FP x FP	0.42	1.22	-97.97	9.28	9.28	9.28	79	44	55.70

Notes: Metallurgical bores (FM); Rotary bores (FD), Exploration pits (PO) – disposed of; Roto-percussive bores (FP); Diff. – difference. 1 There are no pairs with distances. 2 Old pit data disposed of due to location problems and to the data themselves. New due to high grade difference (>20%) to. 3 AN>0.5 x AN<05 – Represents pair counting when one analysis was greater than and the other less than 0.5% Ni. The percentage column is the sample percentage under those conditions, which much be regarded as a project risk.

Source: AMEC, 2006

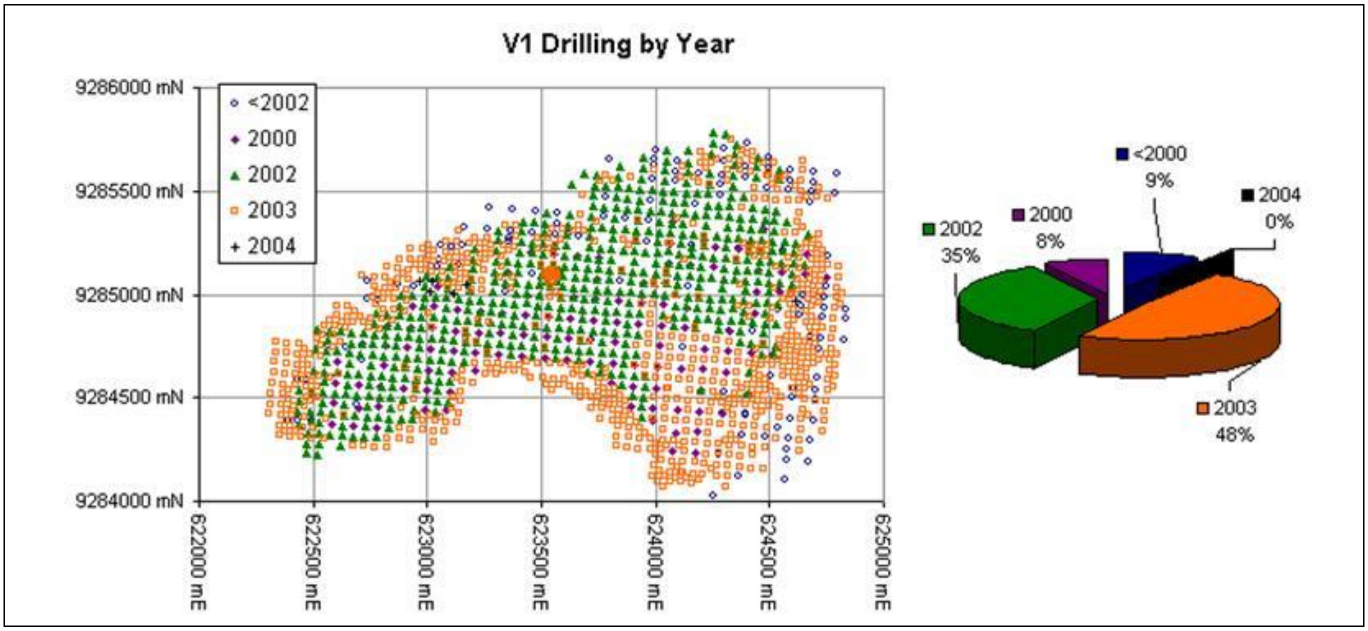
AMEC also compared some of the twin holes and the results were conclusive that no sampling bias occurs, but the comparison between metallurgical holes and others shows a clear bias, explained by the methodology used for sampling. AMEC agreed that metallurgical hole samples should not be used for resource estimation.

10.4 Comments on drilling

10.4.1 Pre-2000 data

Golder Associates (Golder, 2004) reported that data collected prior to 2000 may be unreliable. Snowden interrogated the 2005 Vale resource estimation databases (GCDBV1.mdb and GCDBV2.mdb) and determined that only the V1 deposit was affected by drilling prior to 2000 (Figure 10-6).

Figure 10-6 V1 collar locations coded by year of drilling



Source: Snowden, 2005

The pre-2000 drilling comprises approximately 9% of the V1 collar table with the holes predominantly located on the margins of the deposit. In all cases, the early drilling has been infilled by later programs.

The more current and reliable data sufficiently surrounds the pre-2000 drilling to mitigate any risks posed by the more historical data in the resource model. Snowden therefore considered any risk associated with pre-2000 drilling to be globally negligible and locally minor.

10.4.2 Crossover sampling

Golder reported that samples collected using a crossover sampling bit had the potential to cause smearing of intercepts due to the potential transfer of material down the side of the drill string between the bit and the point at which the sample crosses over to the inner sampling tube of the RC drill string. Snowden could not determine from the available databases, whether Vale had recorded where a crossover bit had been used as opposed to a face sampling bit. Snowden was therefore unable to comment on the spatial extent to which potential downhole smearing occurs.

Snowden commented that although interpreted mineralised boundaries may be offset by downhole smearing, the mineralised thickness interpreted from the results is generally correct albeit offset vertically from the true position. Given that resource estimation of each local block grade is derived from many samples in the local search neighbourhood, Snowden considered any risk associated with crossover sampling to be locally minor and globally negligible.

10.4.3 Cone and quarter sampling

Golder reported that samples collected prior to 2003 were collected using a cone and quartering method which had the potential to introduce poor precision into the resulting subsamples. Snowden proposed that poor precision would be identified in the results of the duplicate field samples collected prior to 2003. Snowden's analysis of Vale duplicate results showed an improvement in duplicate sample precision from May 2003 onwards which may coincide with the introduction of riffle splitting versus quartering. However, while the degree of improvement was significant, the degree of precision for samples prior to May 2003 appeared acceptable for resource estimation purposes.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

HZM has not conducted any sample preparation, analyses and security undertakings on the Vermelho licence as all sample preparation and analyses on the licence was completed prior to HZM reporting the acquisition of the Project in December 2017.

The following is extracted from AMEC (2006) and Snowden (2005), reviewed and amended by the author for inclusion in this Technical Report.

11.1 Sample preparation

The sample preparation was initiated by ALS Chemex at the preparation facility at Parauapebas and was completed at the preparation facility at Luiziania. The general preparation procedure was as follows (crushing only applied to diamond drilling samples):

- Drying at 105°C
- Crushing to 95% passing ¼” (6.35 mm) in a jaw crusher (every 50 samples, mass control on one sample; every 20 samples, granulometry control on one sample)
- Homogenising with four passes and splitting using a Jones splitter with 12 mm openings, to obtain at least 1 kg sample to be submitted for pulverisation (the coarse reject was stored as backup)
- Pulverising with a ring-and-puck pulveriser to 95% passing 150 mesh (every 50 samples, mass control on one sample; every 20 samples, granulometry control on one sample)
- Splitting a 20 g aliquot for assay (the fine reject was stored as backup).

Sample preparation protocols conform to industry standard practices and are suitable for nickel laterite deposits. Checks of preparation specifications are commendable.

11.2 Assaying

All samples were assayed at ALS Chemex in Vancouver, Canada, using the ME-ICP81 method. The following components were assayed: Ni, Al_2O_3, Co, Cr, Cu, Fe_2O_3, MgO, MnO, SiO_2 and Zn. The ME-ICP81 method consists of the fusion of a 0.2 g sample aliquot with 2.6 g of sodium peroxide, digestion in 250 ml of 10% and analysis by atomic absorption spectroscopy (AAS) for Ni and by inductively coupled plasma atomic emission spectroscopy (ICP-AES) for the rest of the components. According to the laboratory, the analytical precision was better than 7.5%. The detection limits are listed in Table 11-1. The official hardcopy assay certificates were stored on site.

Assaying methods are standard for this type of deposit.

Table 11-1 Detection limits of the ME-ICP81 method

Element/oxide	Unit	Upper limit	Lower limit
Al_2O_3	%	30	0.01
Co	%	30	0.002
Cr	%	30	0.01
Cu	%	30	0.005
MgO	%	30	0.01
MnO	%	30	0.01
Ni	%	30	0.005
Zn	%	30	0.01
Fe_2O_3	%	80	0.1
SiO_2	%	100	0.01

Source: AMEC, 2006

11.3 Quality assurance and quality control findings

11.3.1 Golder

A preliminary review of the quality assurance (QA) procedures, with respect to the resource data, was carried out by Golder in April 2003 (Golder, 2003). Golder reported that:

- Drillhole data collected prior to 2000 was somewhat unreliable due to loss of original samples, lack of quality control (QC) samples and poor survey control in some cases; however, Vale has generally replaced the old data locations with a new grid of RC holes which have industry standard QA procedures in place.
- RC drilling and collection procedures for new holes were expected to produce good quality samples albeit with some risk of downhole smearing when crossover drill bits are used in soft ground conditions.
- Prior to 2003, samples were split by coning and quartering on a rubber mat, which is a process that can introduce poor precision into the resultant subsamples. From 2003, samples were riffle split through a Jones riffle splitter and were pre-dried if necessary.
- Golder could not confirm that the bulk density measurements determined from drill core were dry density values.
- Survey control on drillhole collar locations was satisfactory.
- All RC samples collected during 2002 and 2003 were re-assayed for nickel grade because prior x-ray fluorescence (XRF) estimates had been shown to understate nickel concentration.
- No detailed description of the analytical methods was available for review.
- QC checks included field and laboratory assay duplicates, blank and standard reference samples, and checks on pulp particle sizes and masses.
- The basis of certified values for standard reference materials was poorly documented and the standards available at that time did not test high grade nickel and cobalt accuracy.
- High grade assays of magnesia, alumina and iron oxide have been truncated by upper detection limit values of the analytical method.
- 4,698 samples (not affected by truncation) have total oxide values below what Golder considered an acceptable limit of 85% by total mass, where the mass discrepancy is explained by “loss on sample ignition” of hydroxyl, carbonate and sulphate compounds.

11.3.2 Snowden

Snowden visited the Vermelho deposit in December 2003 and completed a QA review of the drill sampling collection methods being utilised (Snowden, 2004a). The key results of this review were:

- Resource definition drilling samples were collected in a consistent and secure manner, and processes were in place to ensure acceptable levels of primary sample recovery
- Subsampling methods followed a logical process of drying weighing and riffle splitting prior to laboratory despatch
- Standard, blank and duplicate samples were included in the sample despatches to the laboratory
- Bulk density values were determined from dried and waxed pit samples using a water displacement method
- A random check of logs from 12 holes found that the quality of Vale geological logging was satisfactory and consistent with the physical samples of stored sample chips
- Sampling issues raised by a prior auditor (Golder) had been or were in the process of being addressed

- The interim electronic resource database that was in place at the time of audit required additional validation and some correction to above detection limit values for magnesia and alumina.

11.3.3 Agoratek

As part of the current review, Vale provided Snowden with a copy of a report from Agoratek International (Agoratek), who was contracted by Vale to independently review QAQC of the Vermelho database in September 2004. Agoratek reported that:

- Sample preparation was performed by ALS in Carajás.
- Assay of the sample pulps was completed by ALS in Vancouver with:
  - XRF determination of oxides on fused pellets
  - Acid digestion and ICP determination for metals (except for nickel).
- Acid digestion and AAS determination for nickel.
- Check assays were analysed at a second laboratory (Gamik).
- Vale project personnel verified the resource database in 2004 and Agoratek concluded that on completion of a two-stage review the data entry error rate for assay data was negligible and estimated to be less than 0.5%.
- Standard reference materials used by Vale were not properly certified and Agoratek subsequently recertified the standards.
- The corrected standard results indicated that several assay batches were suspected to be outside acceptable levels of accuracy and Agoratek recommended that Vale assess the effect of discarding these results for resource estimation purposes.
- A 4.5% relative negative bias (grades understated) was identified for magnesia results from ALS and Agoratek recommended upwards scaling of magnesia results or estimates.
- A positive 19% relative bias (grade overstated) was identified for MnO.
- Blank materials were not certified; however, results indicate a contamination rate in the order of 1.5% of submitted barren materials.
- Submissions to umpire laboratory of Gamik did not include standards with the check assays.
- Only the four main attributes of nickel cobalt, silica and magnesia were reviewed in detail.
- Vale did not visit or acquire monthly reports from the primary assay laboratory.

11.4 Quality assurance and quality control findings

Vale used detailed procedures at Vermelho for every operation, from drilling and sampling to sample preparation and assaying, including recommendations to minimise errors. A QC procedure was implemented during the V1 and V2 exploration programs. The program included the following control operations and control samples:

- Mass control after crushing (2%) and pulverisation (5%)
- Sieve tests after crushing (2%) and pulverisation (5%)
- Coarse duplicates (3%): inserted after splitting the original sample, before the granulometric separation (blind to the laboratory)
- Pulp duplicates (5%): inserted after pulverisation (blind to the laboratory)
- Standard samples: three standards were inserted in the sample batches: a project standard (PJ, 2%), an international standard (PI) and a synthetic standard (PS)
- Check samples (5%): submitted for external analysis to a secondary laboratory (CVRD/Gamik).

AMEC evaluated the Ni and Co coarse duplicate and pulp duplicate data for V1 and V2 (which were reported together), and concluded that the preparation quality was within acceptable ranges, as well as the analytical precision (maximum 10% failures; Table 11-2; Figure 11-1 to Figure 11-4).

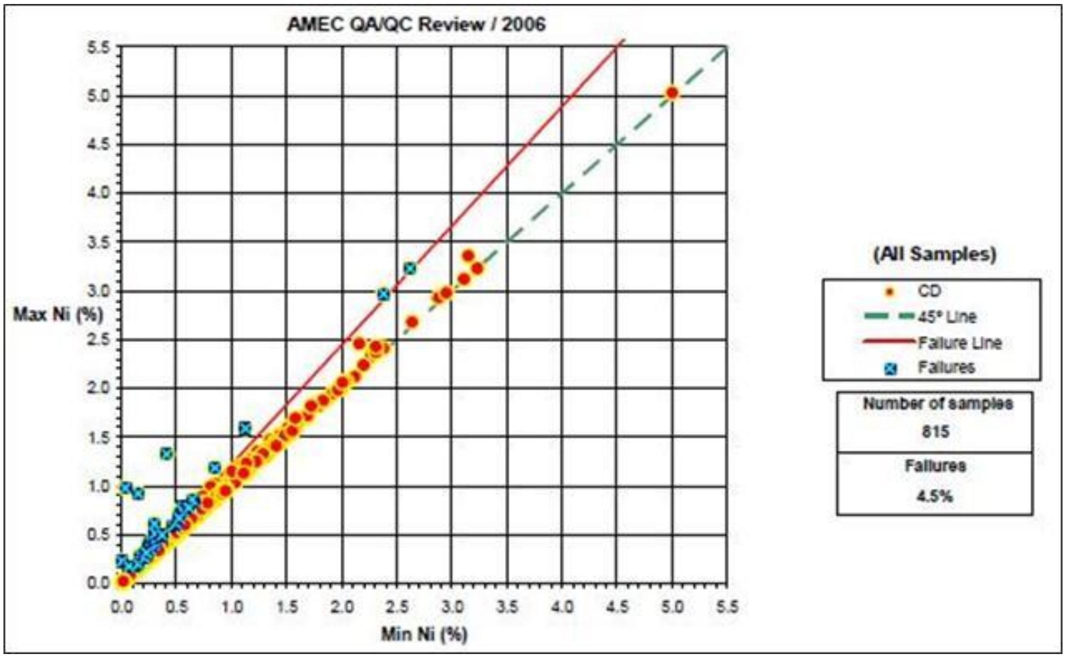
Table 11-2 Duplicate summary

Sample type	Element	No. of samples	No. of failures	Failures (%)*
Coarse duplicates	Ni	830	37	4.5
	Co	830	14	1.7
Pulp duplicates	Ni	8,527	151	1.8
	Co	8,527	331	3.9

**Maximum acceptable proportion of failures is 10%*

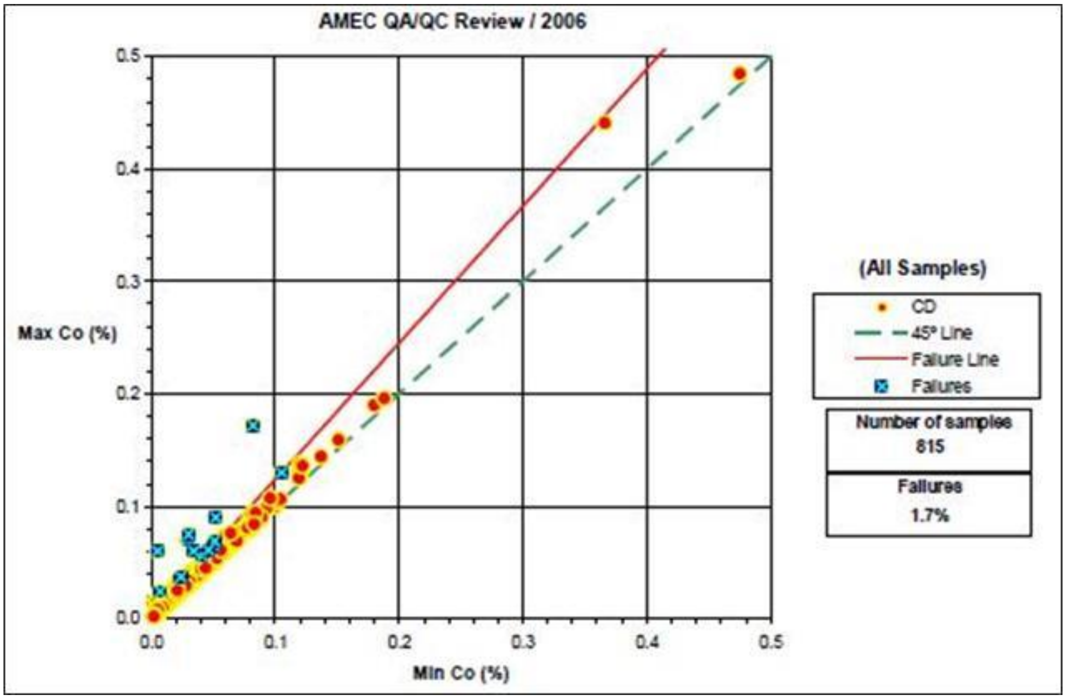
Source: AMEC, 2006

Figure 11-1 Ni in coarse duplicates



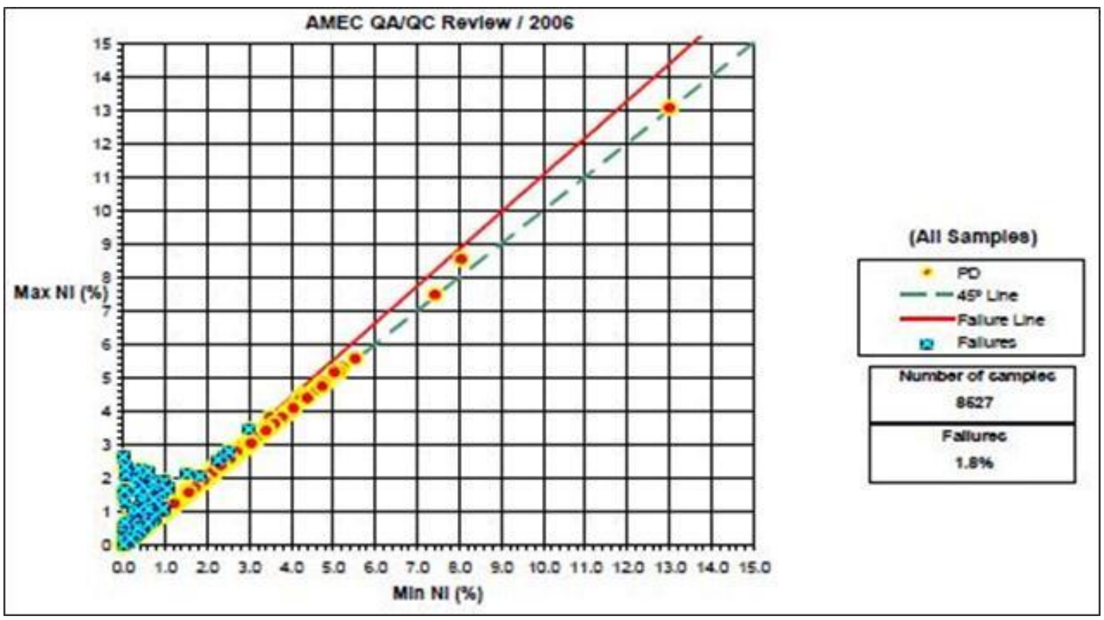
Source: AMEC, 2006

Figure 11-2 Co in coarse duplicates



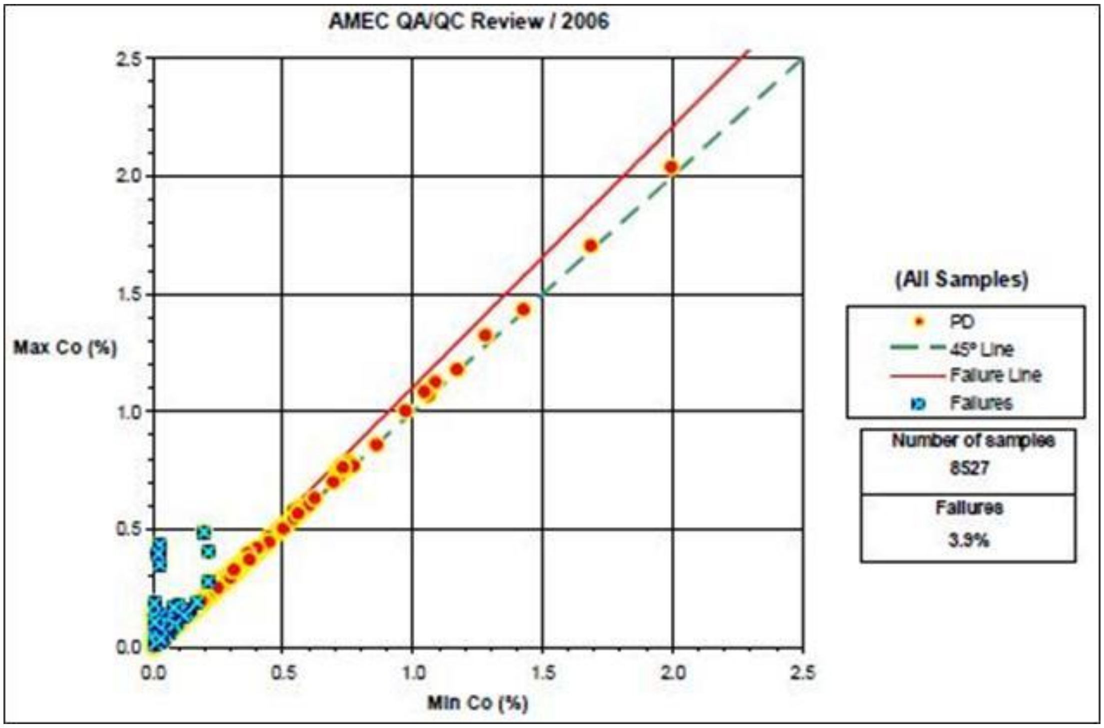
Source: AMEC, 2006

Figure 11-3 Ni in pulp duplicates



Source: AMEC, 2006

Figure 11-4 Co in pulp duplicates



Source: AMEC, 2006

Four project standards were used initially for assessing the analytical accuracy, but five additional standards were prepared and inserted later in the submission batches. The standards included in the first set were certified through a round robin with 15 international laboratories, which assayed each of the four standards 20 times. The standards included in the second set were certified through a round robin with 10 international laboratories, each of which produced four assays for each standard. The certified values for both sets are listed in Table 11-3 and Table 11-4 respectively.

Table 11-3 First set of standards used at Vermelho

Standard	Best value Ni (%)	Control limits Ni (%)	Best value Co (%)	Control limits Co (%)
VRM-1 (BT)	0.335	0.306–0.364	0.048	0.038–0.058
VRM-2 (BT)	0.615	0.574–0.656	0.024	0.017–0.031
VRM-3 (AT)	1.118	1.053–1.183	0.057	0.046–0.068
VRM-4 (MT)	0.909	0.858–0.960	0.106	0.091–0.121

Source: AMEC, 2006

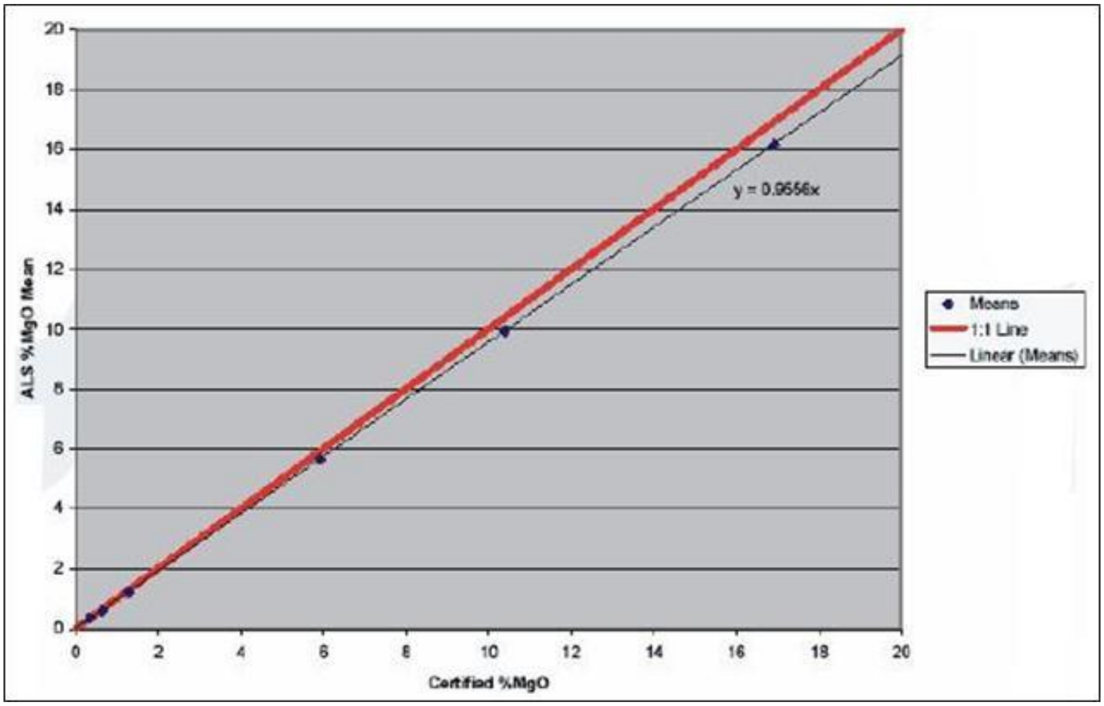
Table 11-4 Second set of standards used at Vermelho

Standard	Element	Mean (%)	RSD (%) (I)*	RSD (%) (II)**
FEBT	Ni	0.635	2.15	1.24
	SiO_2	4.052	4.56	4.3
	MgO	0.351	7.81	7.5
	Co	0.031	5.01	3.86
SIBT	Ni	0.550	2.34	1.91
	SiO_2	62.400	0.84	0.83
	MgO	1.302	2.46	2.4
	Co	0.039	3.93	1.67
PDF	Ni	1.013	2.31	2.01
	SiO_2	29.110	2.31	2.25
	MgO	10.401	1.26	1.23
	Co	0.058	4.59	3.71
AAAT	Ni	2.279	2.19	2.13
	SiO_2	43.950	2.24	2.15
	MgO	16.930	2.32	2.27
	Co	0.045	4.58	1.81
FEAT	Ni	1.205	2.7	2.24
	SiO_2	6.806	2.91	2.53
	MgO	0.652	3.51	3.19
	Co	0.225	4.48	2.69

**Including the Standard Error of the Mean; **Excluding the Standard Error of the Mean.*
Source: AMEC, 2006

Agoratek (2004) reviewed the standard assay data, and prepared accuracy plots (ALS Chemex Mean versus Recalculated Certified Values) for multiple standards. After excluding a certain number of outliers, Agoratek concluded that ALS Chemex did not show any significant relative biases for Ni, SiO_2 and Co, although a -4.5% relative bias was apparent for MgO (Figure 11-5).

Figure 11-5 MgO average vs nest value for multiple standards

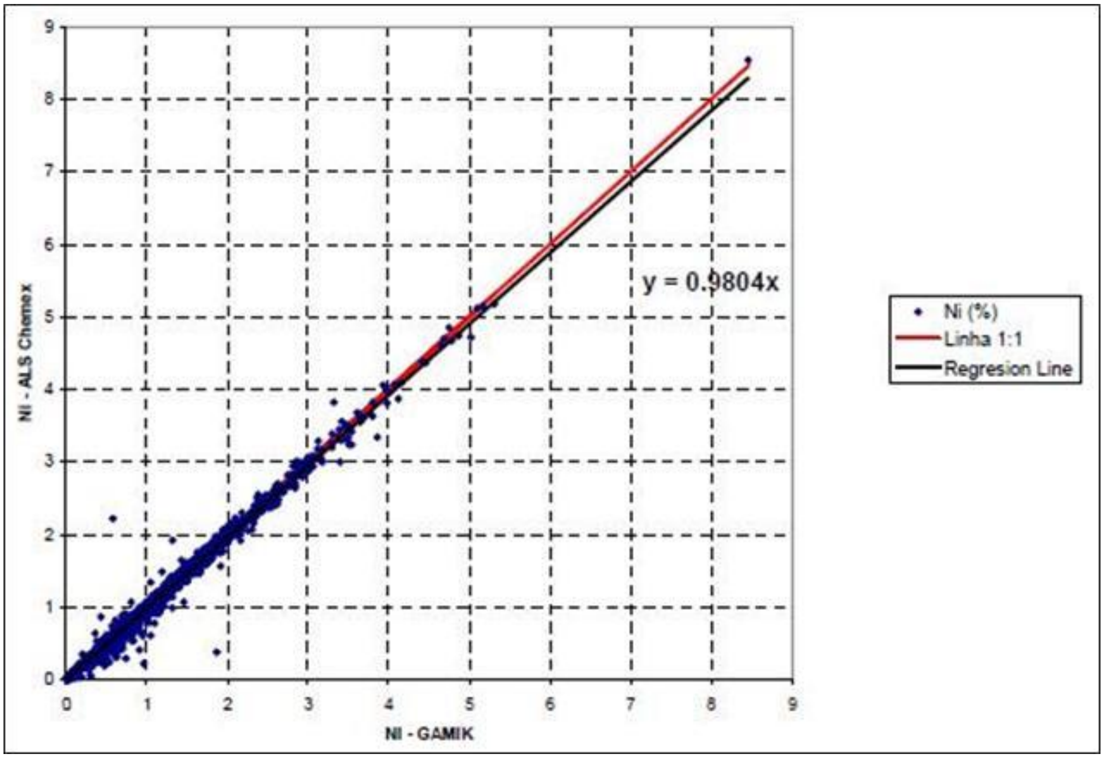


Source: AMEC, 2006

Agoratek (2004) also studied the results of 4,785 check assays performed at Gamik as a secondary laboratory. Their report confirmed the absence of a statistically significant biases for Ni (-2.0%; Figure 11-6) and SiO_2, the presence of a larger, although still acceptable MgO bias (-4.5%) relative bias of ALS Chemex as compared to Gamik, and the presence of a very Co large bias (-19%; Figure 11-7) and a large positive MnO bias (19.5%) of ALS Chemex as compared to Gamik, possibly related to reporting difficulties at Gamik, as well as other minor disturbances (Fe_2O_3, Cr). Agoratek (2004) recommended that the incompatibilities between standard results and check assay results be studied and solved if they would be deemed important for metallurgy.

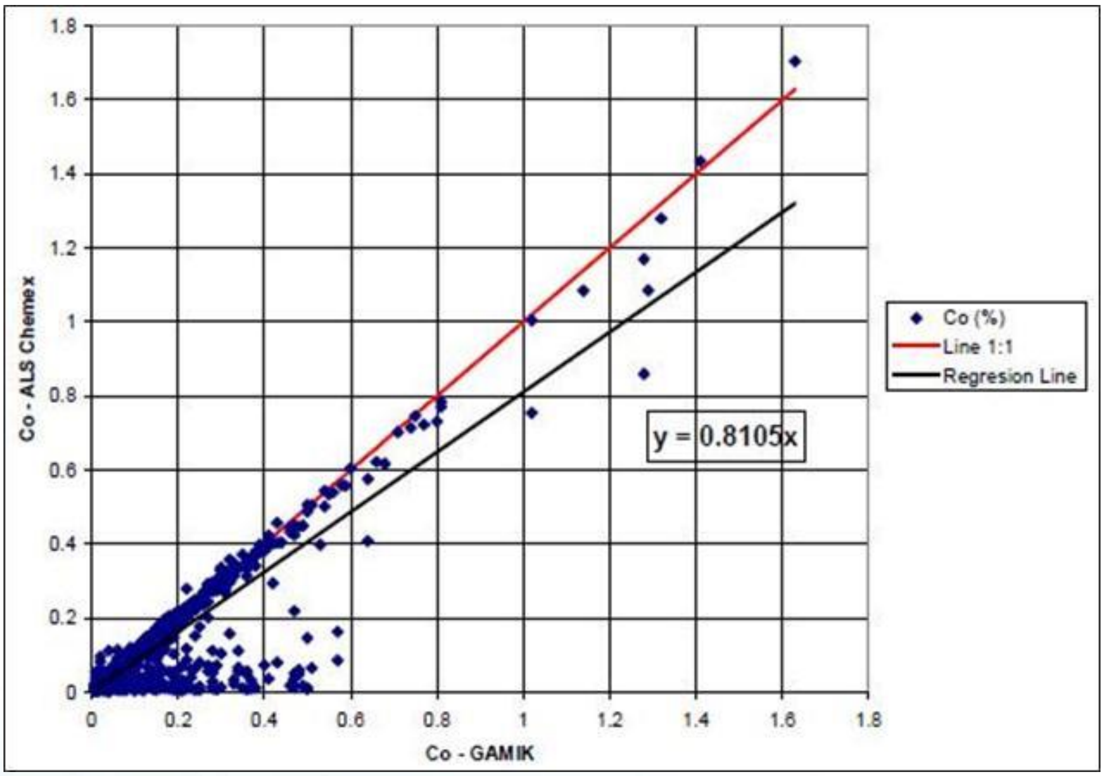
Vale inserted a certain number of coarse blanks during sample preparation at Vermelho. However, the barren condition of those blanks does not seem to have been previously established, which limits the use of such results to assess the possibility of contamination during sample preparation.

Figure 11-6 ALS Chemex vs Gamik – Ni in check samples



Source: AMEC, 2006

Figure 11-7 ALS Chemex vs Gamik – Co in check samples



Source: AMEC, 2006

11.5 Bulk density

As described by AMEC (2006), Vale determined the bulk density at Vermelho through three different methods:

- Water Displacement Method on Drill Core: Bulk density was measured by the water displacement method in selected trimmed sections of intact and coherent core. Core selected for measurement of bulk density by this method was dried, weighed on air (*Wa*), covered with a thin layer of paraffin, and weighed again under water (*Ww*). The *D* density was calculated as follows:
  - $D\ (t/m^3) = W_a / (W_a - W_w)$
- Thin Wall Tube Method: A 15 cm diameter cylinder, with razor-sharp edges, was inserted in the ground (Figure 11-8), after which it was carefully extracted by excavating around the cylinder until it was uncovered. The material recovered within the cylinder was dried and weighed. The bulk density was calculated as the ratio between the dry weight and the inner volume of the cylinder.
- In Situ Tests: Bulk density was calculated as the ratio between the dry weight of the material extracted from a cube-shaped excavation at the bottom of a test pit and the volume of water required to fill the excavation previously lined with a thin, waterproof plastic sack.

Figure 11-8 Insertion of the thin wall tube

Source: AMEC, 2006

The three methods were compared for various lithological types, and no substantial differences were found in their results. This project has a large bulk density database, allowing local estimation of densities. The bulk density values were kriged during the preparation of the 2005 block model. A summary of the bulk density data is presented in Table 11-5.

During the RC drilling program, 4,800 samples for moisture determination were collected. The samples were bagged, sealed, and submitted to the laboratory, where they were weighed, dried and re-weighed. The natural moisture content was determined as follows:

- M (%) = (WW – DW) / WW

where WW is the wet weight and DW is the dry weight.

Density measurement methods and the number of measurements are suitable to estimate the tonnage of individual mineralised lithologies.

Table 11-5 Summary of bulk densities

Lithology	Code	No. of samples	Maximum (g/cm^3)	Minimum (g/cm^3)	Average (g/cm^3)	Median (g/cm^3)
V1						
Soil/Ferricrete	COB	15	2.297	1.278	1.555	1.447
Gabbro	GAB	5	2.941	2.177	2.499	2.45
Pyroxenite	PIROX	14	2.655	1.855	2.2	2.148
Saprolite	SAP	105	2.751	0.807	1.702	1.746
Ferruginous limonite	SAPFE	252	2.518	0.779	1.39	1.385
Siliceous limonite	SAPSIL	215	2.317	0.665	1.405	1.339
Silica	SIL	61	2.487	1.171	1.809	1.764
Pyroxenitic saprolite	SAPIROX	13	2.739	1.43	2.099	1.956
Gabbro saprolite	SAPGAB	10	2.538	1.474	1.747	1.671
Serpentinite	SERP	15	2.584	2.339	2.416	2.409
Weathered serpentinite	SERPINT	26	2.357	1.375	1.995	2.115
Semi-weathered serpentinite	SERPSI	14	2.612	2.231	2.382	2.364
V2						
Soil/Ferricrete	COB	26	2.813	0.956	1.527	1.303
Pyroxenite	PIROX	3	2.537	2.396	2.482	2.514
Saprolite	SAP	156	2.55	0.881	1.634	1.638
Ferruginous limonite	SAPFE	113	1.865	0.883	1.256	1.246
Siliceous limonite	SAPSIL	436	2.585	0.665	1.499	1.437
Silica	SIL	38	2.449	0.826	1.853	2.013
Pyroxenitic saprolite	SAPIROX	11	2.11	0.903	1.432	1.363
Gabbro saprolite	SAPGAB	3	1.475	1.144	1.335	1.385
Serpentinite	SERP	6	2.478	2.322	2.418	2.429
Weathered serpentinite	SERPINT	14	2.477	1.32	2.079	2.121
Semi-weathered serpentinite	SERPSI	3	2.376	2.305	2.354	2.376

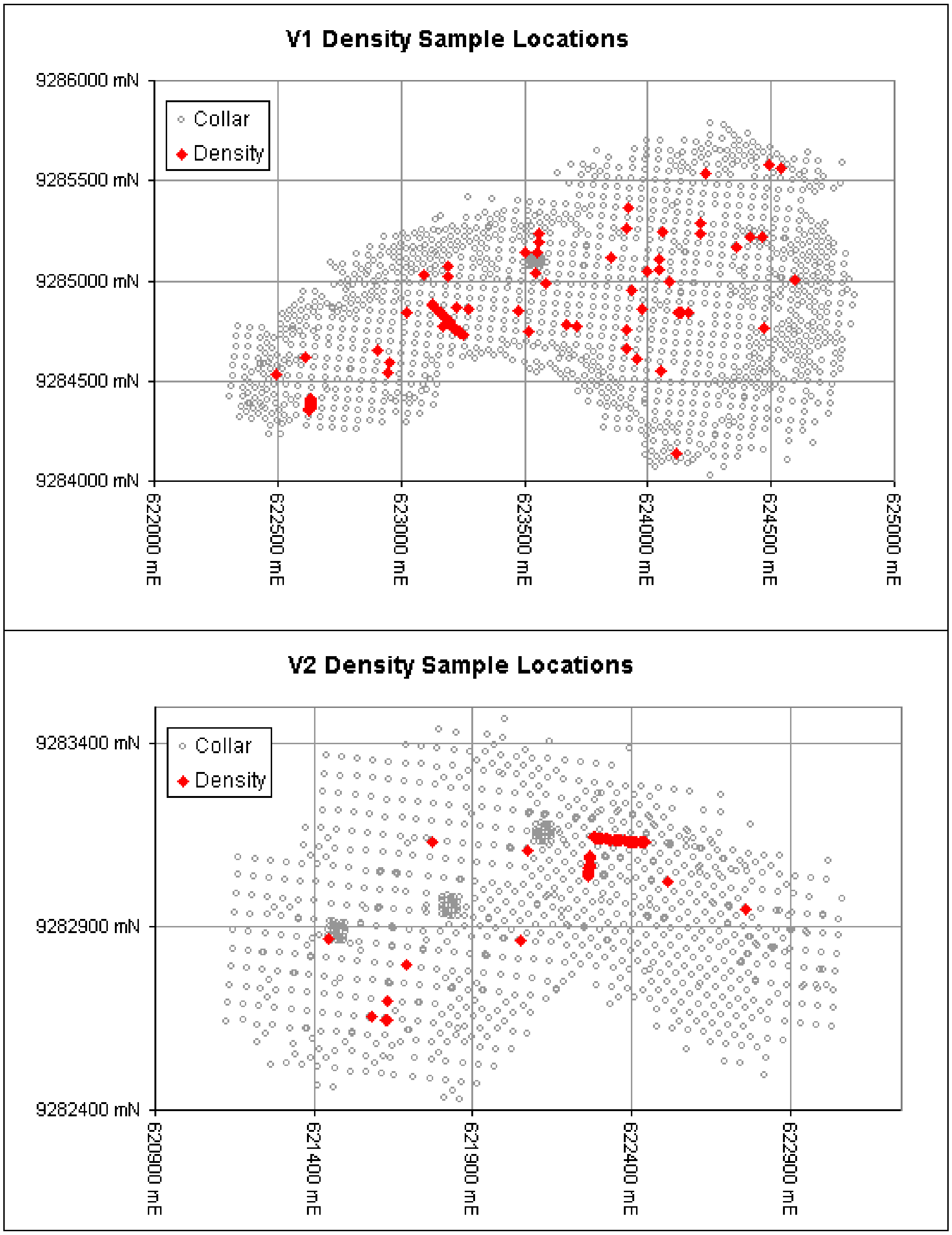
Source: AMEC, 2006

11.5.1 Comments on bulk density sampling (Snowden, 2005)

Golder was unsure as to whether bulk density samples were reported on a wet or dry basis. Snowden confirmed in a later audit that bulk density determinations were carried out on dried and waxed samples. Figure 11-9 shows the spatial location of the available bulk density data which comprises 972 measurements in V1 and 987 measurements in V2.

The apparent clustering of samples was not considered by Snowden to represent a risk to correct density allocation since the bulk density in the model is assigned on a lithology code basis. Importantly, Snowden's experience is that determination of local bulk density in nickel laterite deposits is problematic due to the nature of variation of materials and that while the global estimation of density is reasonable, local estimation accuracy is subject to the correct estimation of material types within each model block.

Figure 11-9 Bulk density sample locations for V1 (top) and V2 (bottom)



Source: Snowden, 2005

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

The author considers that major issues of quality assurance with respect to sampling protocols have been addressed in prior reviews. The site visits by Golder and the author have confirmed that the Vale procedures put in place for collecting, subsampling, and despatching samples generally followed good industry standards.

Snowden's review in 2005 identified other QAQC issues identified in prior reviews that have not all been addressed in subsequent documentation. As such, the author concurs that Snowden has identified the key issues with respect to QAQC and formed an opinion as to the current status of each key issue as follows:

- The pre-2000 drilling may be unreliable: Snowden considers that this data represents only a small portion of the resource database and in most cases, samples have been taken from new holes near these locations using more reliable methodologies.
- Crossover sampling bits may lead to sample smearing downhole: Snowden could not identify how much of the drilling data had been collected using a crossover sampling bit. Snowden considers that in this type of deposit, that any inaccuracy introduced by drillhole wall cavitation would be minor and that local estimates would be acceptable.
- Cone and quarter sampling pre-2003: Snowden's analysis of duplicate data has shown that the cone and quartering method used until mid-2003 has lower sampling precision than riffle splitting but the improvement in precision with riffle splitting is small. However, there is generally poorer precision associated with the samples associated with the central portions of the Vermelho deposits as a function of the focus of earlier drilling programs in these areas.
- Reliability of bulk density values: Snowden considers that Vale has collected sufficient bulk density values to allocated density on a rock code basis. However, Snowden notes that the pattern of sampling for density in V1 is concentrated at the margins on the deposit, while in V2 the samples are more from the core of the deposit.
- Snowden reviewed descriptions of analytical procedures provided by Vale and considers the methods to be of good industry standard.
- Quality of standard materials: An initial review raised concerns as to the quality of standard reference materials and a subsequent study involved re-certification of some standards. Snowden has confirmed these issues and attempted to interpret the available results as discussed further below (see Section 11.6.2).
- Assay truncation (particularly magnesia at 30%): Majority of truncated magnesia values have been re-assayed to determine high concentration values within resource envelope. Snowden considers that the risk associated with remaining capped values is small, because most are outside the resource envelope or are locally constrained by more accurate data.

11.6.1 Sampling precision

Snowden reviewed the representation and precision of the duplicate data for nickel, cobalt, magnesia, silica and iron. The following is noted:

- The field and laboratory duplicate representation is generally good with duplicates collected from nearly all holes in all phases of drilling.
- From April 2003 onwards, the field duplicate level of precision for nickel, silica and iron oxide is good with approximately 90% of the duplicate results having an absolute relative difference in the order of $\pm$10% of the pair mean. However, precision of earlier drilling programs (pre-April 2003) is not as good with approximately 85% of the duplicate results having duplicate pair mean relative difference in the order of $\pm$10% of the pair mean. For magnesia and cobalt, the duplicate precision is lower than that of the other attributes reviewed, with approximately 75% to 85% of duplicate assays having a pair mean relative difference of $\pm$10%.
- The precision of laboratory repeats is very good with nearly 100% of duplicates for nickel, magnesia silica and iron oxide having a pair mean relative difference of $\pm$10%, with only cobalt having marginally lower precision.

Accepting that the average of duplicate values is a more robust estimate of the grade at each sample location, the results for the duplicate data indicate that nickel, iron oxide and silica grades used on the resource estimate are generally within 10% of the mean in approximately 90% of the cases. For cobalt and magnesia, the confidence is lower, with approximately 85% the cases being within 10% of duplicate mean.

11.6.2 Assaying accuracy

Snowden found that the many of the standard reference materials assayed by Vale during the course of resource definition drilling appear to have been incorrectly certified in terms of the certified variability of the data. Additionally, a number of outlier results indicate some sample submission errors and/or heterogeneous standard materials. By removing obvious outlier(s) and an analysis of the mean grades and variability of the standards submitted, Snowden has estimated the levels of accuracy of assaying at the 95% confidence limit for the key attributes to be approximately:

- ±6% relative for nickel grade, with nickel being approximately 1% understated
- ±8% relative for cobalt grade, with cobalt grades being generally accurate
- ±15% relative for magnesia content with magnesia content being approximately 3% understated
- ±7% relative for silica, with silica content generally accurate.

Note that these are subjective estimates due to issues associated with the quality of the standard reference material data; however, the estimates do provide a guide to the level of confidence in individual input assays.

12 DATA VERIFICATION

12.1 Introduction

There have been several data verification studies performed on the primary Vermelho exploration data during its ownership by Vale. The author visited the site in December 2003 and his amended report is included below, followed by other verification studies performed by independent experts.

12.2 Data verification by the Qualified Person

At the request of Vale, Andrew F. Ross of Snowden visited the Vermelho Property on 9 to 12 December 2003. The purpose of the visit was to:

- Observe field procedures during an infill phase of drilling that was underway at the time of the site visit so that the author could endorse the Vale classification scheme with respect to CIM Definition Standards (2014)
- Obtain the latest Gemcom database including re-assayed over limit MgO sample data for ongoing geological and statistical analysis, if requested
- Review the geological basis for partitioning the low-grade nickel grades from the SAP horizon.

The visit was made in the company of Vale's Marcello Costa and included discussions with site geological staff Roberto Albuquerque, Divinio Fleury, Walter Riehl and Marcos Ferreira. Post-visit meetings were held with Vale's Edson Ribeiro and Marcio Fonseca at Belo Horizonte on 13 December 2003 and Vanessa Torres and Ruy Rodriguez at Perth on 19 December 2003.

12.2.1 Drill program

Drilling operations at both V1 and V2 deposits were observed on several occasions during the site visit.

At V2 large diameter (approximately 8-inch) coring operations were observed at three sites: hole FM 105 was in progress of being extended from 50 m to a planned depth of 90 m; rig 121 was in the process of being moved; and the core barrel was being pulled on rig 119, hole FM 78 at 59 m. The author understands that nine rigs are involved in the large diameter core program to obtain samples for metallurgical testwork. The equipment is operated by contractor Geosol on a 2 x 12-hour shift basis for an average advance of 7 m per day.

RC drilling was in progress at V1. Contractor GEOSEDNA's Prospector rig was observed being refuelled at hole 1056 at a depth of 61 m on 9 December. On 11 December, the rig was drilling at a depth of 5 m at line LT725W 175N. Samples were bagged at 1 m intervals.

The accompanying photos (Figure 12-1) illustrate various aspects of the drill programs.

Figure 12-1 Photos from the drill programs (Vale)

Large diameter coring operations – pulling the core barrel at FM 78

Large diameter coring operations – depth markers in core boxes

Infill RC operations – equipment is old but drilling penetration rates are high

Infill RC operations – Prospector W750 articulated rig at V1 LT725W

Large diameter coring operations – safe storage of drill consumables and recycling

12.2.2 Chain of custody of samples from rigs to sample farm

The author observed the safe transport of core from the drill rig to the core logging shed adjacent to the Project Office building, and the safe transport of RC samples from the drill rig to the sample farm on several occasions. The Construtora Zag Ltda (ZAG) geological technicians were responsible for safe transport of the samples.

12.2.3 Sample storage, processing and security

The author found that security in terms of core, and the orderly processing of RC samples, was very good.

The large diameter core boxes were laid out for geological logging and sampling, and these activities were underway at the time of the site visit. After logging and sampling the boxes were stacked, awaiting the results of assays before despatch to Australia for metallurgical testwork. The core required compositing and then transfer to metal or plastic containers to meet quarantine approval.

RC samples were processed in a compound within walking distance of the Project Office. Samples were dried, weighed, and split using a modified riffle splitter recommended by D.F. Bongarcon of Agoratek. The author observed the handling and splitting of several samples and the procedures for decontamination and bagging. The ZAG technicians processed the samples in accordance with the procedures described by Vale staff.

Representative material was cast onto wooden “rules” for geological logging and an additional sample was archived in plastic cups for reference. These are satisfactory ways for storing reference material and the library appeared to be secure and well organised.

Images from the sample preparation process are shown in Figure 12-2.

Figure 12-2 Photos of sample preparation (Vale)

Bagged and labelled RC samples at drill pad, ready for transport to farm

RC sample weighing

Drying of RC samples

Preparation of “rules” for logging of RC chips

Storage of RC chips for reference – lid was removed from box

Logging and sampling of large diameter cores

12.2.4 Density measurements

A laboratory was established in the RC sample compound to handle density measurements of samples taken from the pits (pocos).

The technicians described the procedures for drying and wax coating of samples. The author observed the water replacement method in process and concluded that the procedures conformed to standard industry practice.

12.2.5 Sample preparation and assay

The quartered RC samples were transported to the nearby assay laboratory at Carajás. At the time of the site visit, there were no samples from Vermelho being processed and the author did not review this aspect of the field operations.

12.2.6 Status of Golder's recommendations

Vale had previously engaged Golder to conduct a review of the geology and resource database in April 2003 and the resultant recommendations were considered by Vale. The author discussed the status of Golder's recommendations with Vale staff.

12.2.7 Data provided by Vale

Vale supplied the author with the following data:

- The latest unvalidated Gemcom sample database as two versions, date stamped on 9 and 12 December 2003
- Original logs for 11 holes from line LT700W, deposit V2: V2-FP185; V2-FP-178; V2-FP-179; V2-FP-173; V2-FP-60; V2-FP-114; V2-FP-53; V2-FP-48; V2-FP-50; V2-FP-55; V2-FP-59
- 117 original assay electronic assay certificates, dated from 2002 and 2003
- Source files for duplicates (19), standards (10) and survey (10) results
- Drillhole location maps for V1 and V2 (scale 1:5,000)
- Drill section LT700W section.

12.2.8 Findings

Drill operations

The author observed that field operations were conducted in a safe and professional manner, that drill sites were clean, with samples securely boxed and labelled, or bagged and labelled appropriately in a consistent way.

The author understood that GEOSEDNA was required to re-drill a hole where recoveries fell below 60%. Recoveries were determined by Vale's geological contractors, ZAG, who undertook routine weighing of samples.

It was standard Vale practice for ZAG personnel to monitor RC drill performance and to transfer samples to the sample preparation areas. However, there was no requirement for ZAG to continually remain at the drill site during sampling operations. The author recommended that, in future programs, consideration is given to continuous monitoring by a geotechnician to ensure that sample bagging remains error-free.

Resource database

Some validation checks of the interim database were made by the author and inconsistencies were noted where over-limit MgO and Al_2O_3 assays had been merged incorrectly. These were mostly rectified by site personnel; however, there were still several records where the new over-limit MgO assay values were inconsistent with the earlier values and sample identification switching was suspected.

Further validation of the resource database was recommended, prior to commencement of the FS resource estimates and after all the results had been obtained.

Geological logging

The author selected 12 RC drillholes from line LT700W at V2 for a review of the geological logging and database integrity.

Vale was able to recover the reference material for all requested holes and the author then compared the geological logs and Gemcom lithocodes with the actual chips. The quality of logging and database entries were found to be satisfactory.

Geological surfaces

A low-grade nickel population was identified in the saprolite horizon in Snowden's earlier review of the resource model. The merits of applying a 0.5% Ni basal boundary to screen out the lower grade population was considered; however, it appears that a substantial number of low-grade samples occur as low-grade pockets at higher levels in the horizon. The author recommended applying a 0.3% to 0.5% Ni boundary to screen out included fresh rock both at the base and at higher levels in the profile and then compare this with the bedrock interpretation as determined solely from geological logging.

The author recommended that the selection of surfaces is considered after all the results of the geological mapping are to hand. The geological domaining should then be developed in parallel with contact analysis to determine vertical and lateral grade trends.

Consideration should also be given to the introduction of a transition horizon between the ferruginous limonite and saprolite horizons where MgO grades appear to crossover from extremes of low to high grades.

12.3 Data verification by Snowden

Extracted from Snowden 2005 (Final Feasibility Study Resource Review).

12.3.1 Resource Model Review Process

Snowden staff (C. Standing) reviewed the V2 resource model in Vale's office at Belo Horizonte during 6 to 12 August 2004 and then reviewed the revised V2 model from 23 to 25 August 2004. The V1 model (dated September 2004) was reviewed In Snowden's Perth office from 20 September to 1 October 2004.

The major aspects of the review for the V2 and V1 resource models included:

- Reviewing the geological interpretation in cross-section and as 3D solids and surfaces
- Checking coding of input data against the wireframe models
- Checking coding of block models against the wireframe models
- Review of the statistical analysis of input data for domain verification and consideration of grade estimation methodology
- Review of the variogram analysis for verification of variogram and search parameters

- Checking the output files from the kriging runs
- Checking for uninformed blocks and that grades are within expected limits
- Checking the application of the Gemcom script files used for dilution, classification and merging of the component folders
- Visual validation of the grade, density and classification models
- Checking the integration of the component grade and density data into the final diluted and undiluted resource models
- Examination of the validation trend plots of the grade models by easting, northing and elevation provided by Vale.

QAQC data and graphs for duplicate standards were briefly examined. These were not reviewed in detail as it was understood that Dr D. Francois-Bongarcon had undertaken a detailed review of all the QAQC data and was preparing a report that covers this aspect.

Following the initial review of the V2 model, a few issues were identified which were summarised in a PowerPoint presentation that was given to the Vale's resource team in Belo Horizonte with Snowden's understanding of the actions that were to be taken by Vale. Majority of Snowden's recommendations and issues were addressed and the revised V2 resource model was provided to Snowden on 19 August 2004. Following a check that the major issues had been addressed, this model was exported for the mining study.

The recommendations that were not addressed by Vale are as follows:

- Following statistical analysis, it was noted that some Fe (25 values) and MgO (96 values) grade data have been increased by capping of grades at the lower end of the Fe distribution. It was expected that this would be revised; however, while this grade capping is not advisable, this is not regarded as significant as it affects less than 0.2% of the data.
- The search orientations for the ferruginous limonite domain were revised but the variogram parameters were not.
- The nugget effects for Co and MnO used for grade estimation were too low and should have been increased.

It is understood that these issues are minor and that, as the development of the V1 model was regarded as a priority, these issues were not addressed. Snowden considers that they will have had an insignificant impact on the resource model.

It was understood that 1 m data compositing would be implemented for V1. This was not undertaken, but this is not regarded as being significant as it affects less than 2% of the data.

During the review of the V1 model, the undiluted model was developed, and corrections were made to the Cu grade model and the waste density model by Vale. The V1 model was then exported for the mining study. During this time, a Zn grade model was provided by Vale for V2 that was exported and incorporated into the V2 mining study.

12.4 Data verification by AMEC in 2006

12.4.1 Database integrity review

Vale implemented an in-house database management system at Vermelho. Logging and sampling data were manually entered into the database. The database system had built-in algorithms for checking overlapping or missing values. The remainder of hand-entered data was checked visually, using the original data sheets. The survey data were digitally entered from the survey equipment, with no backup hard copies. The assay data were digitally entered by the laboratory, but official, backup hard copies of assay certificates were retained and filed.

Drillhole collars

AMEC inspected the collars of 16 drillholes from V1 and 14 drillholes from V2 (1.2% and 1.6%, respectively, of the drillholes included in the database), and measured the coordinates with a Vista e-Trex global positioning system (GPS). Most collars were well conserved and identified, with marked long wood sticks. The measured coordinates were compared with the database coordinates, and the maximum planar difference found was 28.3 m, within the current GPS precision (Table 12-1 and Table 12-2).

Table 12-1 Review of V1 collar coordinates

Hole ID	Database coordinates		AMEC coordinates		Differences	
	X (m)	Y (m)	X (m)	Y (m)	X (m)	Y (m)
V1FD024	623,247.000	8,284,717.000	623,262.042	9,284,716.239	15.0	-0.8
V1FP117	622,996.000	9,284,740.000	623,014.047	9,284,735.795	18.0	-4.2
V1FP148	623,698.000	9,284,731.000	623,715.914	9,284,725.655	17.9	-5.3
V1FP171	623,605.000	9,284,847.000	623,626.049	9,284,834.127	21.0	-12.9
V1FP181	623,562.000	9,284,801.000	623,570.475	9,284,788.588	8.5	-12.4
V1FP250	623,291.000	9,284,806.000	623,272.407	9,284,814.508	-18.6	8.5
V1FP263	623,101.000	9,284,820.000	623,072.703	9,284,831.045	-28.3	11.0
V1FP279	623,058.000	9,284,782.000	623,069.499	9,284,784.505	11.5	2.5
V1FP389	623,759.000	9,284,738.000	623,765.789	9,284,722.573	6.8	-15.4
V1FP540	624,205.000	9,284,885.000	624,232.755	9,284,881.131	27.8	-3.9
V1FP650	624,111.000	9,284,798.000	624,122.873	9,284,791.138	11.9	-6.9
V1FP667	624,216.000	9,284,330.000	624,233.445	9,284,783.603	22.3	10.6
V1FP706	623,967.000	9,284,768.000	623,968.933	9,284,758.824	12.8	-8.0
V1PO240	622,892.000	9,284,657.000	622,904.758	9,284,649.004	12.8	-8.0
V1PO241	623,015.000	9,284,840.000	623,023.997	9,284,836,969	9.0	-3.0

Source: AMEC, 2006

Table 12-2 Review of V2 collar coordinates

Hole ID	Database coordinates		AMEC coordinates		Differences	
	X (m)	Y (m)	X (m)	Y (m)	X (m)	Y (m)
V2FD019	622,257.000	9,283,095.000	622,268.548	9,283,094.617	11.5	-0.4
V2FP108	621,537.000	9,282,662.000	621,527.983	9,282,657.086	-9.0	-4.9
V2FM064	622,736.000	9,282,810.000	622,747.226	9,282,802.205	11.2	-7.8
V2FM099	621,617.000	9,282,761.000	621,634.927	9,282,747.595	17.9	-13.4
V2FP125	621,415.000	9,282,673.000	621,473.649	9,282,610.940	12.8	-7.7
V2FP212	622,046.000	9,282,967.000	622,056.227	9,282,962.007	10.2	-5.0
V2FP265	621,464.000	9,282,609.000	621,473.649	9,282,610.940	9.6	1.9
V2FP271	621,567.000	9,282,658.000	621,578.284	9,282,653.147	11.3	-4.9
V2FP273	621,519.000	9,282,715.000	621,532.458	9,282,706.406	13.5	-8.6
V2FP331	621,574.000	9,282,853.000	621,595.280	9,282,851.717	21.3	-1.3
V2FP342	621,789.000	9,282,835.000	621,794.791	9,282,834.179	5.8	-0.8
V2FP487	622,997.000	9,282,939.000	623,005.684	9,282,930.887	8.7	-8.1
V2FP853	621,479.000	9,282,878.000	621,483.931	9,282,873.348	4.9	-4.7
V2FP860	621,451.000	9,282,915.000	621,462.615	9,282,913.209	11.6	-1.8

Source: AMEC, 2006

As part of the collar review, AMEC also plotted 204 drillhole collars from V1 and 105 drillholes from V2 (14.8% and 12.0%, respectively, of the drillholes included in the database) on the final version of the topographic maps, with 5 m contour lines, and compared the collar altitude with the location altitude for each hole, according to the contour lines represented in the topographic map. None of the holes presented significant differences in elevation, and all appeared to be correctly plotted.

Original logs vs database entries

The original drillhole logs were properly filed on site and could be easily reviewed. The original logs included both coded and detailed handwritten descriptions of the lithology, as well as codes for primary and secondary minerals and information about structures.

AMEC checked three fields (From, To and Lithology) of the database against 50 original logs from V1 and 42 original logs from V2, corresponding to 3.6% and 4.8%, respectively, of the drillholes included in the database, to determine the entry error rates. AMEC also reviewed, by double data entry, the Ni assay data of 2,768 samples from V1 and 2,892 samples from V2.

Table 12-3 lists the entry errors found in the review. The partial and total entry error rates were acceptable, given the industry standard of less than 1% error for databases supporting resource estimates.

Table 12-3 Entry errors in the V1 and V2 database

Field	No. of entries	No. of errors	Error rate (%)
V1			
From–To	543	1	0.2
To	543	4	0.7
Lithology	543	4	0.7
Ni	2,768	2	0.1
V2			
From	287	0	0.0
To	287	3	1.0
Lithology	287	2	0.7
Ni	2,892	0	0.0
Total	**8,150**	**16**	**0.2**

Source: AMEC, 2006

Sampling consistency

AMEC reviewed the sample intervals as stored in the database and compared the sample material with the original logs from 50 drillholes from V1 and 42 drillholes from V2 (3.6% and 4.8% of the holes, respectively, included in the database). AMEC checked the main lithologic types and the sample intervals, which were adequately recorded. The sample data were properly identified in the sample bags and tags. Sample intervals (From–To) agreed with major lithologic changes on drill logs and respected the main lithologic boundaries.

Geological interpretations

AMEC reviewed all the geological sections from V1 (48 sections) and from V2 (38 sections), located at 50 m intervals, and reviewed the geometry of the interpreted geological shapes in the sections. Additionally, AMEC reviewed the original logs from 20 holes from V1 and 20 holes from V2 (1.4% and 2.3% of the holes, respectively, included in the database), and visually cross-checked them against the information displayed in the corresponding geological sections. AMEC also checked the original logs versus the RC chips from 13 holes from V1 and 17 holes from V2 (0.9% and 1.9% of the holes, respectively, included in the database), and then compared these to the remaining core of two diamond drillholes, one from each deposit.

AMEC recognised that the interpretation respected the data recorded in the logs and the sections, as well as the interpretation from adjoining sections, and was consistent with the known characteristics of this deposit type. The lithologic model had been diligently constructed in conformance with industry standards practices. AMEC did observe that in some sections, the interpolation rules applied to ferruginous saprolite bodies that pinched out were not consistent (the pinching out distance was irregularly established).

In general, geological interpretations were suitable to support resource estimates.

Conclusions

AMEC made the following conclusions:

- Vale used very detailed and accurate written procedures to guide practically every aspect of the geological work
- Drilling, logging, sampling, and geological interpretation of the V1 and V2 deposits were conducted consistently and diligently

- Database integrity had been favourably tested; the database entry error rates were within acceptable ranges
- The QAQC protocols in application by Vale for sampling, sample preparation and assaying were generally adequate: precision, accuracy and contamination were regularly determined
- AMEC processed the laboratory duplicate assays for Ni and Co and confirmed that the preparation variance and the analytical precision for these elements appeared to have been within acceptable limits.

AMEC reviewed the analysis prepared by Vale and Agoratek for the analytical accuracy based on inserted standards, and confirmed the absence of significant biases for Ni, SiO_2 and Co, and a relative larger, although still acceptable, bias for MgO (4.5%):

- However, the check assay results yielded contradictory results: a larger, although still acceptable MgO bias (-4.5%) relative bias; and substantial Co bias (-19%) and MnO bias (19.5%) of ALS Chemex as compared to Gamik were apparent
- Despite the above-mentioned facts, AMEC considered that the assay data from these exploration campaigns could be used for resource and reserve estimation purposes
- The density determination procedures applied by Vale at V1 and V2 were adequate and correspond to industry standards.

12.5 Data verification by HZM in 2016 and 2017

12.5.1 2016 HZM due diligence

HZM advised the author of due diligence items completed by HZM during one of its site visits over the period 21 to 23 September 2016:

- Drillhole logging from eight core holes were checked
- HZM checked that drillhole database conformed to assay certificates on four selected holes with significant SAP intersections
- HZM checked that core and RC chippings and rejects are available in the storage sheds
- HZM compared two selected DDH and twin RC holes.

HZM advised the author that all the checks were satisfactory.

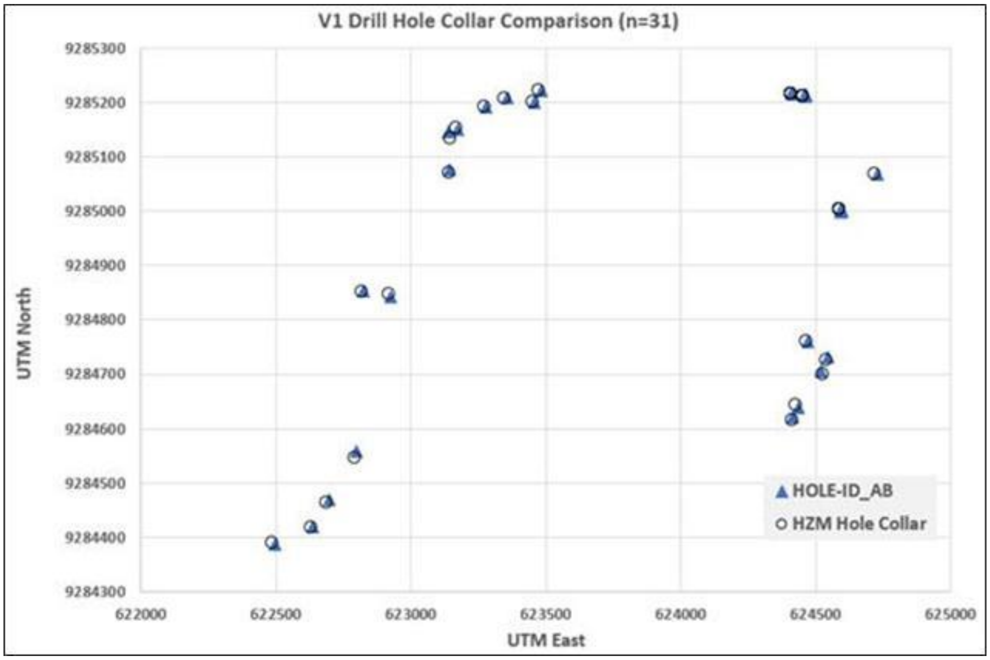
12.5.2 2017 HZM due diligence

Between 17 and 20 October 2017, an HZM team visited Vermelho targets V1 and V2 to locate and measure coordinates for DDH and RC drillholes, pits, outcrops, and access tracks. Coordinates were measured with a handheld Garmin GPS map 62S instrument configured for the SAD69 datum.

A total of 66 drillholes, monitoring wells and pits were located, 37 at V1 and 29 at V2. Of this total, 54 drillholes were identified in the Vermelho drillhole database and were used in the comparison. The database coordinates are also based on the SAD69 datum.

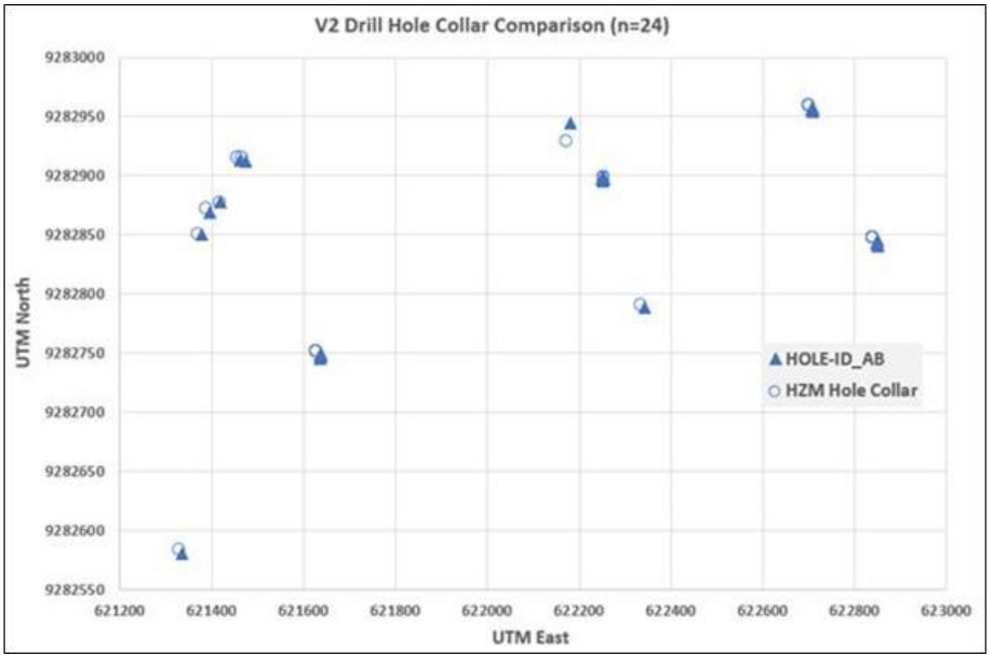
Of the 55 drillholes, 31 are at target V1 and 24 are at target V2 (Figure 12-3 and Figure 12-4).

Figure 12-3 Location of verified drillholes at V1



Source: HZM, 2019

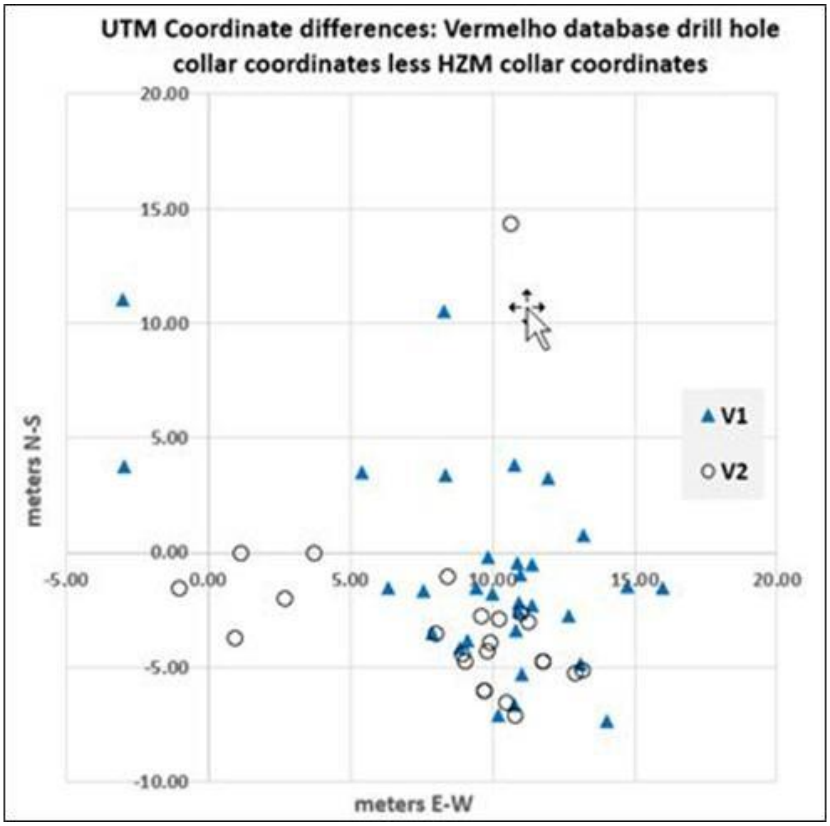
Figure 12-4 Location of verified drillholes at V2



Source: HZM, 2019

Figure 12-5 shows the absolute difference in metres between the coordinate pairs. HZM collar coordinate measurements are, on average, 9.2 m east and 1.8 m south of the Vermelho collar coordinates identified in the Vale drillhole database.

Figure 12-5 Difference in collar coordinates



Source: HZM, 2019

The measured coordinates compared to the database coordinates showed a maximum planar difference of 16 m, considered to be within GPS precision. Based on this evaluation the drillhole database coordinates are considered by HZM to accurately reflect the location of the drillholes.

12.6 Verification in 2019

The author undertook a site visit on 5 August 2019 in the company of HZM staff. The access tracks are largely overgrown. HZM has transferred all the diamond drill core and bulk metallurgical samples to its new core storage facility which will support the Project during the next phase of work.

12.7 Qualified Person's opinion on the adequacy of the data for the purposes used in the technical report

The field operations used to acquire the data for Vermelho Mineral Resources were checked several ways so that a view of data integrity could be formed by the author. Drilling and sampling procedures appeared to be satisfactory at the time of the site visit, and the author relied on the earlier observations of Golder with respect to the quality of survey control and collar locations.

The author conducted checks on the consistency of Vale's geological logging from a key section from V2, and this was found to be satisfactory.

The author concluded from his observations and Golder's report that the field procedures to collect data are satisfactory for a CIM Definition Standards (2014) compliant MRE.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Project was first developed by Vale with the objective of becoming its principal nickel-cobalt operation. Extensive work was undertaken on the Project at Scoping (PEA), PFS and FS stages. This included drilling and pitting programs totalling 152,000 m, batch and full-scale pilot testwork and detailed engineering studies. The Project was subsequently taken through a Feasibility program with Vale reporting a positive development decision in 2005. The Project was designed around the construction of a HPAL plant to process the nickel/cobalt laterite ores. The FS included a five-year metallurgical testwork and pilot plant program which demonstrated 96% average leach extraction rates of nickel and cobalt. In addition, LME grade nickel (metal) cathode was produced. The FS proposed production capacity was 46,000 t/a of metallic nickel, and 2,500 t/a of metallic cobalt, with an expected commercial life of 40 years. The Project was subsequently placed on hold after delivery of the FS due to Vale liquidity issues.

Process flowsheet development for nickel laterite deposits is dependent upon the metallurgical characteristics (both chemical composition and physical form) of the laterite mineral types (limonite, transition, and saprolite). To correctly characterise the material, it is necessary to carry out the appropriate metallurgical testwork and through this, establish an appropriate flowsheet amenable to the processing of those specific ores. Selection of a suitably representative sample set from the main lithologies in each deposit (and the associated mineral resource) is key to this process.

It should be noted that the testwork conducted, and presented in this Technical Report, on the Vermelho deposits has been conducted by Vale and its consultants and reported in GRD-Minproc (2005) prior to its acquisition by HZM. In late 2018 to early 2019, HZM conducted limited laboratory-scale testwork on selected samples from the Vermelho deposit to validate the potential to produce high-grade nickel and cobalt sulphate products using the HPAL process route with further downstream purification and recovery stages. This testwork is described in Section 13.5. The following is extracted from the GRD-Minproc study (2005), reviewed and amended by the author for inclusion in this Technical Report.

Table 13-1 Glossary of some abbreviations used in this section

Abbreviation	Description
AR	Ammonia re-leach
EW	Electrowinning
HPAL	High pressure acid leach
LME	London Metal Exchange
MCP	Mixed carbonate precipitate
MHP	Mixed hydroxide precipitate
MSP	Mixed sulphide precipitate
PAL	Pressure acid leach
SX	Solvent extraction

13.2 Process testwork – Prefeasibility Study

Metallurgical testing at the PFS stage was designed to cover all project options to be evaluated financially and was comprised of two main programs:

- A pilot plant program for process development and engineering data collection
- A bench-scale program for evaluation of variability within the orebody, specifically ore behaviour for beneficiation, leaching, settling and slurry rheology.

The pilot plant program was performed at Lakefield Oretest, Australia from November 2002 to March 2003, and the variability testwork was performed concurrently at Vale's Mineral Development Centre, located close to the main project office. Cross-checks of bench-scale test procedures and chemical analysis between laboratories were performed throughout the program.

13.2.1 Sampling

Significant attention was spent on ensuring that the samples used for the testwork were representative. The following procedures were adopted:

- Both bench-scale and pilot programs used large diameter drill core samples
- Drilling was distributed geographically throughout both deposits and covered major ore types and lithologies (refer to Section 7)
- The composition of the overall sample was adjusted to have the same chemical composition of the LOM as per the Scoping Study mine plan
- The variability program used 10% of the 2,500 samples available, with 1 m collected from every 10 m of the mineralised laterite profile, which ranges typically from 40 m to 80 m. In this way, samples for the pilot and bench testwork were interrelated and represented the same geological domains.

The drill core samples were composited according to their crystalline silica content into four large volume samples for pilot and process development testwork. These were:

- "LowSi" sample, representing mainly a ferruginous limonite ore type (SapFe), comprised of limonite with a small amount of silicates or quartz
- "HighSi" sample, representing mainly a siliceous limonite ore type (SapSil), comprised of approximately 50% relatively coarse SiO_2 (as quartz) with the remaining mass as fine limonite
- "MidSi" sample, representing an intermediate ore type (SapFeSi), comprised of approximately 35% coarse SiO_2 with the remaining mass as fine limonite and fine SiO_2
- A blended sample, composed of amounts of the three samples above, and included all major ore types.

Small amounts of the saprolite ore type (garnierite), found in the contacts and immediately below the main ore types were blended within the samples, to simulate some blending of the oxidised ore with a low Mg garnierite that will occur at the mine.

The composition scheme of the samples allows direct correlation of pilot plant and variability samples, since each of the bulk pilot samples tested has its variability counterparts. Also, subsamples of the pilot bulk composites were used for the same bench-scale tests in the variability study, to assess scale-up and validate the test methodology.

13.2.2 Pilot testwork

The pilot plant program was designed with consideration of pressure acid leaching of limonitic laterite nickel ores. This is a commercialised process, with pilot plant design and operating experience reported from four functioning industrial plants (at the time; there have been several more constructed since). Therefore, pilot-scale testing was deemed sufficient for collection of process engineering data, and a demonstration scale plant was not deemed necessary as scale-up and equipment design was based on existing industrial operations.

The design of the pilot plant program also considered that the complexity of process flowsheet and related engineering, and that the variation between laterite ores requires a significant amount of pilot testing for new projects to quantify process parameters and to collect engineering data.

To ensure collection of the required data for a robust PFS report, equipment suppliers and reagent vendors, namely Ciba, Cognis, Delkor, Eimco, Filtres Philiipe-RPA, Larox, Ondeo-Nalco, Outokumpu (now Outotec) and SNF-Floerger participated in the pilot program to collect data for specific reagent selection and equipment sizing.

Pilot testing comprised four beneficiation tests (to remove free silica using crushing, screening and cyclones) and three HPAL runs with a total of 21 days of operation: the HPAL-MHP circuit operated for 14 days and the PAL-MSP circuit operated for seven days. The MHP circuit pilot plant included all unit operations including production of nickel cathode; whereas the MSP circuit pilot plant produced mixed nickel-carbonate sulphide that was refined at bench scale to produce nickel powder[3].

Overall, the technical viability of processing Vermelho ore by a PAL-based process route was confirmed by the pilot plant program. No fatal technical flaws were observed in the beneficiation step or in any of the hydrometallurgical flowsheets that were evaluated, namely:

- HPAL/MHP/AR/SX/EW
- HPAL/MHP/AR/MCP
- HPAL/MHP/AR/SX/Precipitation/Calcination
- HPAL/MSP
- HPAL/MSP/POX/SX/Hydrogen Reduction.

Testwork results indicated favourable ore behaviour, with good rheological characteristics, high nickel and cobalt extractions, low relative acid consumption, high refining recoveries and a positive response to both the MHP and MSP processing routes.

Table 13-2 presents the chemical assays of the pilot plant products. LME specifications were achieved for metallic nickel and cobalt products.

Table 13-2 Pilot plant products[4]

Element	Nickel (cathode)		Intermediate products		Metal products (green briquettes[5])		MCP (ppm)	$NiCo_3$ bulk precipitate (ppm)	NiO powder (ppm)	CoS precipitate (ppm)
	LME spec. (ppm)	Vale product (ppm)	MHP (%)	MSP (%)	Nickel (ppm)	Cobalt (ppm)				
Al	-	5	0.24	0.40	15	25	50	<5	25	200
As	100	5	-	0.01	-	-	<5	<5	<5	135
Ca	-	15	0.13	0.01	5	20	60	80	186	700
Co	1,500	75	2.43	3.22	235	99.89%	10,850	7.5	15	35.7%
Cr	-	2	0.01	0.02	3	5	12	<2	4	30
Cu	200	2	0.16	0.12	2	20	1,870	2	3	605
Fe	200	83	0.18	3.60	35	100	95	10	40	250
Mg	-	2	1.6	0.02	8	3	460	40	67	16.7
Mn	50	1	4.2	0.004	0	5	4,205	6	10	25
Ni	99.98%	99.99%	38.3	54.6	99.95%	525	41%	46.4%		1,170
Pb	100	25	-	-	-	-	<2	<2	1	37
S	100	20	4.25	37.3	130	290	1.85%	300	171	37.1%

[3] Nickel metal that is produced by the hydrogen reduction of the nickel sulphate/ammonium sulphate solution (i.e. Sheritt process).

[4] Not all of these products are proposed for production at the Project.

[5] The green briquettes are nickel powder that has been formed into a briquette, normally the briquette is sintered before being dispatched, the term "green" normally means before sintering.

Element	Nickel (cathode)		Intermediate products		Metal products (green briquettes[5])		MCP (ppm)	$NiCo_3$ bulk precipitate (ppm)	NiO powder (ppm)	CoS precipitate (ppm)
	LME spec. (ppm)	Vale product (ppm)	MHP (%)	MSP (%)	Nickel (ppm)	Cobalt (ppm)				
Si	50	10	0.46	0.24	20	0	110	50	20	250
Zn	60	2	0.82	0.72	3	0	9,770	10	26	8.9%
NiO	-	-	-	-	-	-	-	-	99.5%	-

Source: GRD-Minproc, 2005

13.2.3 Batch variability testwork

Beneficiation

Batch-scale variability tests were conducted at the Vale Mineral Development Centre. Tests were conducted on 220 individual drill core samples of 6” x 1 m, weighing approximately 20–25 kg each on a dry basis. Each sample was crushed to -2” and a split was taken for scrubbing at 55% solids, 45 revolutions per minute (rpm) and six minutes residence time (in the rotating drum). The scrubbed material was then screened at ¼” (6 mm), 32# (0.5 mm), 100# (0.15 mm) and 200# (0.74 mm) mesh fractions. Each fraction was assayed for Al, Co, Cr, Fe, Mg, Mn, Ni, and Si. Element upgrades and mass recoveries were assessed for the 100# and 200# fractions.

The main outcome of the variability program was the demonstration of the predictability of upgrading behaviour by geology and chemical composition, as well as the applicability of the models based on the variability testwork to predict results of pilot plant tests.

Leaching

Fifty beneficiated samples were selected for leaching variability testwork. This was aimed at verifying leach extractions and acid consumption for individual samples. The main findings of this program were the minimal variation of nickel extraction between the samples tested, as well as excellent predictability of acid consumption as a function of chemical composition.

13.3 Process testwork – Final Feasibility Study

As the process route for nickel production was fully defined in the PFS, testwork at the FFS was aimed at further optimising and demonstrating, in longer-term campaigns and under steady-state conditions, the complete processing flowsheet. This objective was to provide consistent data for the engineering design and to minimise risks.

The availability of consistent ore-specific process data and the experience of the design team relating to lessons learned in full-scale nickel laterite operations were considered key by Vale when contracting and designing the FFS testwork. In this way, personnel with previous operational experience in nickel laterite plants was deployed by GRD-Minproc and Vale at the contracted laboratory (SGS Lakefield Oretest) to oversee plant operations and participate in all testwork related meetings.

13.3.1 Integrated pilot testwork

The integrated pilot testwork took place from 16 April to 2 October 2004, and included beneficiation, HPAL, precipitation, nickel refining and cobalt refining circuits.

Sample composition

Samples for the testwork were obtained by large diameter diamond drilling within the area delineated by Vale as the 10-year pit for both the V1 and V2 deposits. The drillhole locations were selected by the Vale Geology team and an external consultant, Dominic Bongarçon (Agoratek). It was intended to ensure sample representivity and to further refine the beneficiation equations that would be used to predict the Ni and Mg grades that would be fed to the autoclave.

A total of 6,640 x 1 m interval samples was delivered to SGS Lakefield Oretest for pilot testwork, with approximately 550 inter-twinned samples sent to the Vale Mineral Development Centre for batch variability testwork.

The 1 m interval samples were blended to supply eight bulk samples for beneficiation testwork. The criteria used for generating the beneficiation composites was to obtain different compositions regarding silica and Mg content, to cover expected extremes in the mine planning.

The beneficiation products were then used to compose the feed samples for four HPAL to nickel refining campaigns. The composition was completed with differing proportions of each beneficiation product and aimed to provide four distinct samples with Ni and Mg grades similar to the 10-year mining plan. In this way, the pilot testwork could capture any mineralogical variability, as it simulated the formation of blending stockpiles in the industrial operation.

Testwork concept and objectives

The testwork concept for FFS aimed to demonstrate the Vale MHP flowsheet in a steady state, using continuous runs for samples representative of the first 10 years of operation. While finetuning and optimisation of the conditions for the selected unit operations was still be undertaken, most operational parameters had already been obtained during the PFS testing phase (refer to Section 13.2).

The goals for the pilot plant campaign were:

- Process and operational demonstration at high plant availabilities and uptime for extended periods.
- Address any recommendations and issues highlighted from the previous pilot campaigns and/or PFS.
- Demonstration of main controls and operating strategy for all key unit operations.
- Collection of additional data for design criteria and equipment specifications for the FFS (recoveries, reagent consumption, product grades).
- Confirmation of beneficiation upgrade relationships.
- Collection of samples for additional vendor testwork, coordinated with pilot plant operation, such as and operational disruptions were minimised. The following vendors and consultants attended the pilot plant: GLV, Outokumpu, Westech, Delkor, Filtres Phillipe, Consep, Corrosion Services Pty, Wah Chang, Lightnin, Ekato, Philadelphia Mixtec, Cognis, Nalco, SNF Floerger, Ciba and Rheochem.
- Monitoring of recycles and build-ups of nickel, cobalt and impurities throughout the circuit over an extended period.
- Additional monitoring of scaling and corrosion parameters throughout the circuit.

The pilot plant was designed and operated with the intention to maximise circuit integration and minimising the effect of circuits being filled, metal tenor build-ups, pregnant leach solution (PLS) surge productions, as well as any semi-continuous steps.

Beneficiation circuit

Beneficiation was achieved by the removal for free silica. Eight samples (approximately 12.5 t dry basis each) were subjected to individual beneficiation runs. After beneficiation, samples were processed in parallel units of Eimco-Deep cone and Outokumpu paste thickeners for the campaigns, to allow a comparative evaluation of their performance.

Four samples were also submitted for hydro-cyclone tests, to collect data for engineering design, since the industrial plant uses cyclones for the finest size classification.

High-pressure acid leach plant and MHP/effluent and residues treatment

The HPAL plant and MHP circuit was fully continuous and incorporated all the seeding and circuit recycles, as well as on-stream effluent and residues treatment.

Four campaigns were performed:

- A four-day optimisation run prior to the first 14-day run to fill the downstream circuits and verify plant mechanical reliability and process control capabilities
- Continuous operation during three runs of 28 x 12-hour shifts at average availability of 98%.

A 70 L autoclave was used. A total of 1,070 hours of continuous operation was completed over a nine-week period, processing 21.4 t (dry solids) of slurry to produce in total 1,055 kg of quality MHP cake containing 221 kg of nickel and 10.5 kg of cobalt.

Nickel refining

The MHP produced in the three upstream pilot plant demonstration campaigns was be processed in parallel and continuously in three runs of 26 x 12-hour shifts.

Nickel refining incorporated all unit operations from ammonia re-leach (AR) to nickel EW, allowing recycle of the raffinate for leaching and the precipitation of cobalt sulphide in a bleed stream. The circuit also included stages of copper and zinc removal.

A total of 696 hours of operation was completed in the three runs processing all the MHP produced in the upstream processing area, to produce 83.6 kg of LME-grade nickel cathode (the residual nickel contained in MHP remained in the circuit inventory).

Cobalt refining

Due to the limited cobalt content in the ore, the cobalt sulphide feed was stored from the three nickel pilot campaigns to provide a sufficient amount for the cobalt refining pilot campaign, incorporating all unit operations from acid re-leach of cobalt sulphide to cobalt EW.

13.3.2 Variability testwork

Bench-scale variability

Variability testwork was predominantly aimed at providing more data to support the mass and metal recovery equations through the beneficiation HPAL feed upgrade circuit.

The bench-scale variability testwork program included 368 additional samples, aimed at providing a larger database for the beneficiation equation correlation, and to enhance the representivity of the model. The samples processed in the FFS phase exhibited the same behaviour as the previous testwork, thus allowing the use of all 550 results for the development and validation of the final correlations in the beneficiation equations.

The variability program also processed composite samples to confirm the smoothing effect in variability due to blending of multiple 1 m, 20 kg individual samples into larger composites (mining blocks).

Additional variability acid leach testwork was also performed to supplement data from previous testwork.

Rheological variability was also assessed by Rheochem, using samples from bench-scale and pilot testwork.

13.3.3 Crushing testwork

Crushing testwork using large volume samples was performed using industrial scale MMD crushers installed at the Igarape Bahia site. These bulk samples were taken by excavating existing road faces and ore outcrops. Additional laboratory crushing characterisation was also performed to assess expected feed size distribution to the beneficiation circuit.

13.4 Process testwork – current Project

The previous PFS and FFS testwork programs completed in 2002 and 2004 respectively were focused on the HPAL/MHP/AR/MCP flowsheet option. The current project has adopted a HPAL/MSP/POX/SX/crystallisation flowsheet (i.e. incorporates a different "back end" of the plant). However, much of the process data from piloting and batch variability work is still directly applicable, including:

- Crushing
- Beneficiation (drum scrubbing, attritioning, screening and hydro-cyclones)
- HPAL
- Partial neutralisation (also referred to as slurry neutralisation)
- MSP
- POX.

In late 2018 to early 2019, HZM commissioned limited laboratory-scale testwork at SGS Lakefield (Ontario) to evaluate the further processing of nickel and cobalt to produce higher purity products, as discussed in Section 13.5.

13.4.1 Beneficiation

The amenability of the Project ores to beneficiation is a critical attribute of the Vermelho deposit and significantly enhances the Project economics. The purpose of the beneficiation is to upgrade nickel and cobalt by rejection of silica. Processing a higher-grade material through the hydrometallurgical plant results in a reduction of the unit operating costs. Due to its economic importance and close dependence on ore mineralogy, a detailed assessment of ore response to beneficiation has been undertaken.

The assessment adhered to the following lines of investigation:

- Assessment of beneficiation technical feasibility by processing four large volume samples in a pilot plant in the PFS stage, and further piloting with eight ore blends in the FFS stage
- Investigation of the relationship between beneficiation performance and mineralogy, by processing a number of drill core samples by batch testwork and assessing the influence of ore geology and geochemistry in beneficiation results
- Assessment of process predictability by correlating pilot and bench-scale test results, with the development of adjustment factors where necessary.

The technical feasibility of the beneficiation process has been proven by piloting programs at Lakefield Oretest during the PFS and FFS phases of work. Twelve large volume samples weighing 10 t to –15 t were processed continuously through a pilot-scale scrubbing and classification circuit. Table 13-3 presents the beneficiation results from both PFS and FFS and is directly applicable to the current Project flowsheet. The FFS pilot beneficiation feed and discharge composition of the eight ore blends is shown in Table 13-4.

Table 13-3 PFS and FFS pilot beneficiation results

Run no.	Sample	Ni (%) feed	SiO_2 (%) feed	Mass recovery (%)	Ni grade (%)	Ni recovery (%)	Ni upgrade (%)
1 PFS	Blended Feed	1.07	38.7	51.6	1.45	70.1	136
2 PFS	Low silica	1.12	8.0	69.4	1.15	71.1	103
3 PFS	Mid silica	1.20	35.3	51.0	1.61	68.6	134
4 PFS	High silica	1.08	52.3	40.0	1.79	66.5	166
1 FFS	Blend 1	1.05	40.6	49.6	1.67	78.2	159
2 FFS	Blend 2	1.02	55.2	37.4	1.90	69.9	186
3 FFS	Blend 3	1.23	28.7	75.3	1.44	88.9	117
4 FFS	Blend 4	0.87	63.0	40.7	1.59	75.0	183
5 FFS	Blend 5	1.11	45.9	48.6	1.68	74.5	151
6 FFS	Blend 6	1.21	42.2	54.3	1.80	81.6	149
7 FFS	Blend 7	0.98	51.9	45.7	1.68	78.9	171
8 FFS	Blend 8	1.29	54.0	45.4	2.22	79.2	172

Source: GRD-Minproc, 2005

Table 13-4 FFS pilot beneficiation feed and discharge composition

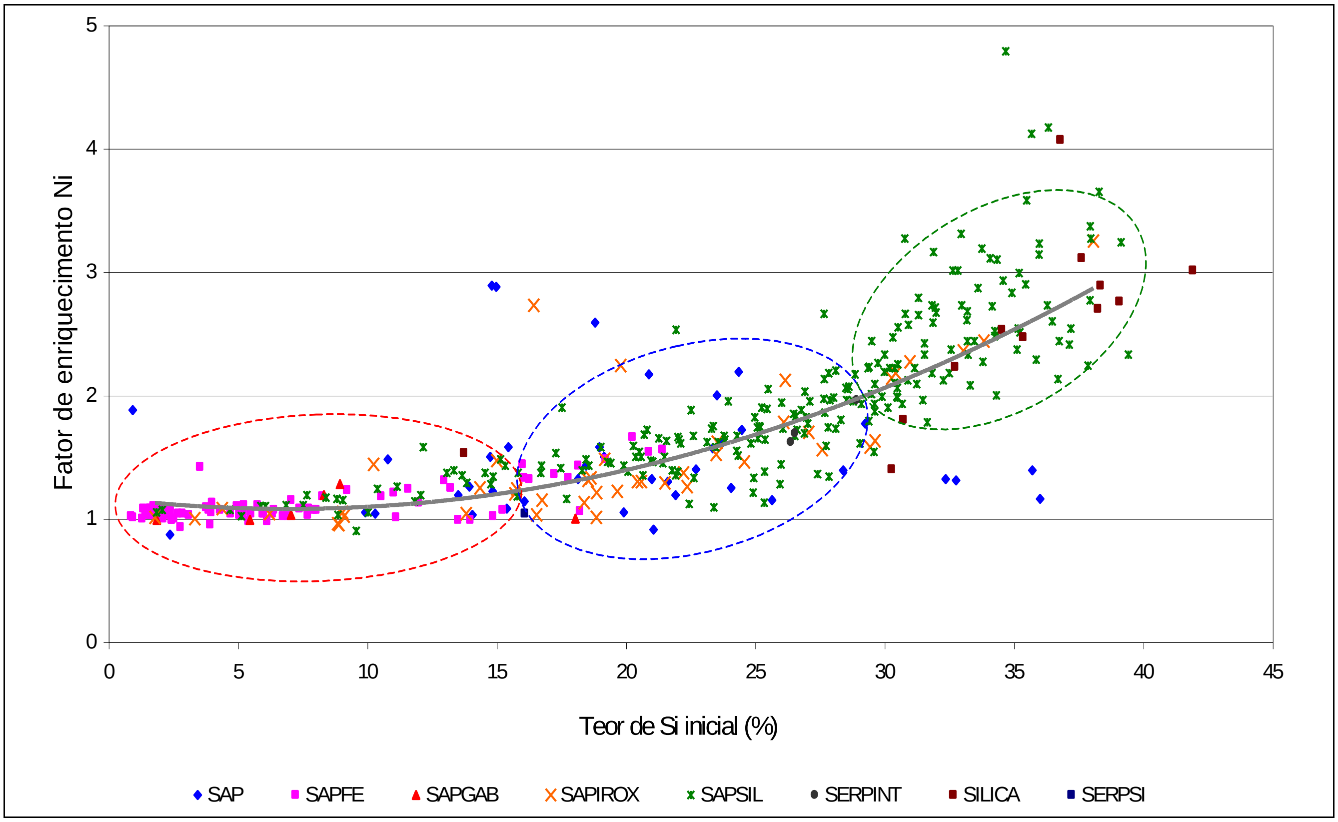
Item	Description	Compositional value							
		Blend 1	Blend 2	Blend 3	Blend 4	Blend 5	Blend 6	Blend 7	Blend 8
Moisture	% w/w	27.2	22.7	28.7	19.4	29.4	28.0	25.0	22.9
Solids SG	Feed	2.94	2.97	2.95	3.05	3.17	3.37	3.16	3.11
	Discharge	3.28	3.41	3.67	n/a	3.66	3.76	3.65	3.65
Ni %	Feed	1.02	1.02	1.16	0.81	1.15	1.29	1.01	1.29
	Discharge	1.61	1.90	1.42	1.61	1.68	1.82	1.78	2.22
Co %	Feed	0.03	0.04	0.11	0.05	0.06	0.09	0.04	0.06
	Discharge	0.05	0.07	0.09	0.08	0.07	0.10	0.07	0.08
Fe %	Feed	17.73	17.61	35.11	16.96	22.72	28.06	18.81	18.56
	Discharge	28.60	32.45	42.37	34.64	36.56	42.34	35.08	34.98
Mg %	Feed	10.09	3.77	2.06	1.70	5.17	2.75	3.68	2.83
	Discharge	8.44	5.36	1.69	2.68	5.94	3.19	5.73	4.76
Al %	Feed	0.75	0.49	1.35	0.45	0.66	0.74	0.51	0.49
	Discharge	1.17	0.88	1.41	0.82	0.95	0.93	0.81	0.79
Mn %	Feed	0.15	0.14	0.52	0.23	0.22	0.45	0.18	0.19
	Discharge	0.20	0.24	0.42	0.34	0.27	0.47	0.28	0.30
Cu %	Feed	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Discharge	0.03	0.02	0.02	0.01	0.02	0.02	0.02	0.02
Zn %	Feed	0.02	0.03	0.03	0.02	0.02	0.03	0.02	0.03
	Discharge	0.03	0.06	0.04	0.04	0.02	0.04	0.04	0.05
Si %	Feed	19.09	26.92	14.15	30.32	21.97	18.17	22.98	23.62
	Discharge	11.80	12.40	7.02	14.68	11.29	8.41	11.49	12.36
Cr %	Feed	0.70	0.99	2.10	1.04	0.61	1.33	1.13	0.97
	Discharge	0.54	0.90	1.26	1.07	0.58	1.09	0.84	0.96

Source: GRD-Minproc, 2005

As can be seen in Table 13-3, the beneficiation process involving ore scrubbing/washing and screening, results in nickel upgrade values of up to twice that of the ore feed grade. The process separates and removes coarse crystalline (chalcedonic) silica and chromite particles from fine ferruginous limonite. The pilot results confirm that upgrade and mass recovery is dependent on silica in feed, and therefore the upgrades factors are dependent on ore mineralogy.

A total of 550 batch variability tests from both the PFS and FFS test programs were completed on the beneficiation process. These tests have shown a consistent relationship between the upgrade ratio, and silica content of feed, among the main ore types, as presented in Figure 13-1. All lithotypes (and consequently ore types) are seen to follow the same upgrading behaviour whereby the improvement on nickel grade increases with increasing silica content.

Figure 13-1 Nickel upgrade as a function of lithotype and silica content



Source: GRD-Minproc, 2005

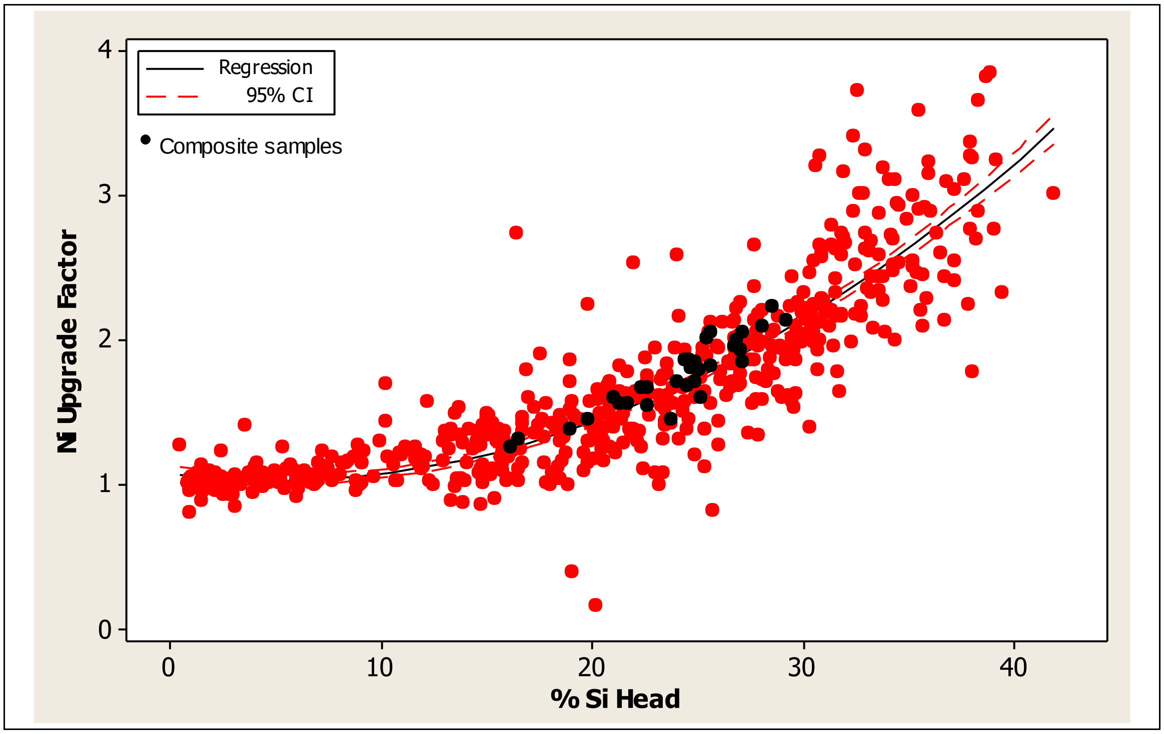
Since the rejection of free silica particles is the main driver of beneficiation behaviour, Si-based equations to represent average ore upgrade were directly derived from testwork results by Vale Mineral Development Centre in Brazil and verified by Geostats in Perth (Geostats, 2004). These equations were used as inputs for mine planning in the FFS and are shown in Table 13-5. These equations were then validated against pilot plant results which provided an independent dataset and confirmed the relationships with a high degree of confidence. The variability and composite samples compared to predicted, are shown in Figure 13-2 (nickel upgrade) and Figure 13-3 (mass recovery).

Table 13-5 Beneficiation upgrade equations based on silica content

Item	R^2	Average upgrade equation for <150 µm
Ni	0.799	$Up\ (Ni)_V = 1.083 - 0.01828*Si + 0.001786*Si^2$
Co	0.453	$Up\ (Co)_V = 0.9975 - 0.01913*Si + 0.001663*Si^2$
Mg	0.575	$Up\ (Mg)_V = 1.018 - 0.01362*Si + 0.001238*Si^2$
Fe	0.826	$Up\ (Fe)_V = 1.092 - 0.02168*Si + 0.001915*Si^2$
Al	0.651	$Up\ (Al)_V = 1.147 - 0.05443*Si + 0.002883*Si^2$
Si	0.439	$Up\ (Si)_V = 1.029 - 0.03714*Si + 0.000621*Si^2$
Cr	0.425	$Up\ (Cr)_V = 0.8939 - 0.03727*Si + 0.001779*Si^2$
Cu	0.590	$Up\ (Zn)_V = 1.171 - 0.04988*Si + 0.002740*Si^2$
Zn	0.547	$Up\ (Cu)_P = -1.237 + 0.2348*Si - 0.004481*Si^2$
Mn	0.689	$Up\ (Mn)_V = 1.092 - 0.03133*Si + 0.002034*Si^2$
Mass	0.703	% Mass Recovery (% < 100#) = 84.66 – 1.772*Si

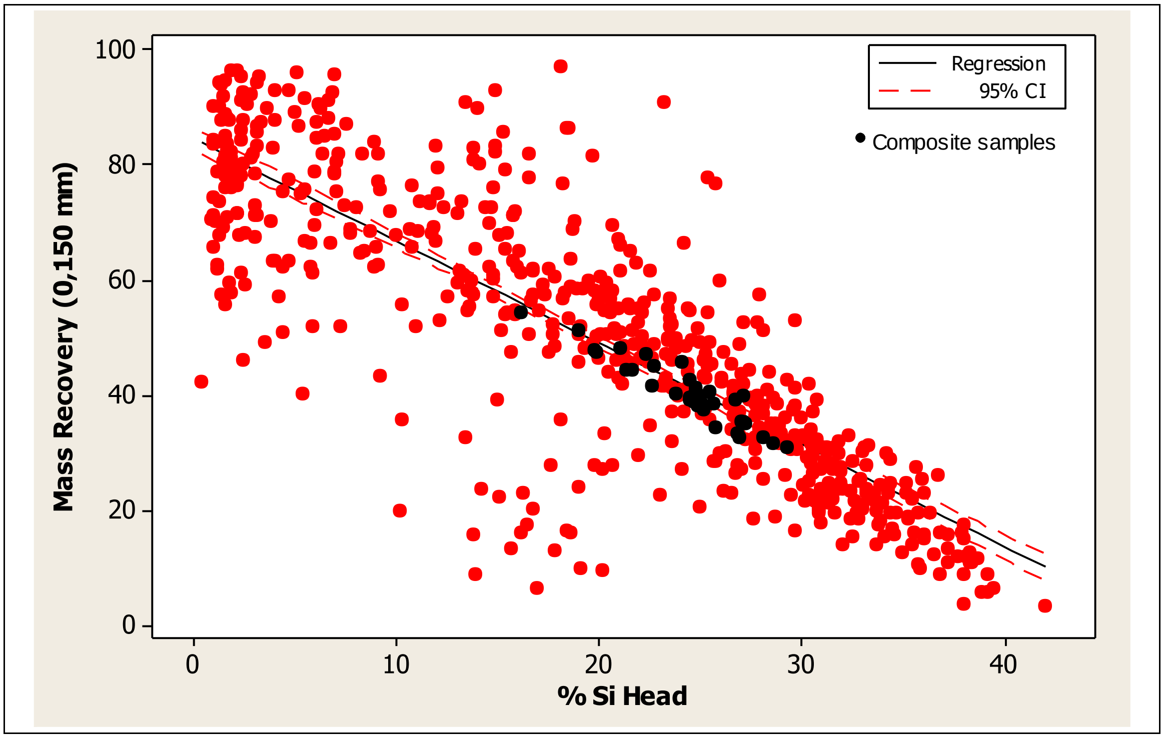
Source: GRD-Minproc, 2005

Figure 13-2 Nickel upgrade for variability and composite samples compared to predicted



Source: GRD-Minproc, 2005

Figure 13-3 Mass recovery for variability and composite samples compared to predicted



Source: GRD-Minproc, 2005

13.4.2 High-pressure acid leach

An initial set of 50 HPAL variability batch tests were completed with samples that could be correlated with the PFS pilot program. Tests were conducted at Lakefield Oretest. Samples tested represented nickel grades ranging from 0.7% to 5.7% and Mg grades ranging from 0.3% to 13.6%.

Average nickel and cobalt extractions for samples tested in batch variability tests were 97.5% and 95.0% respectively, corresponding to terminal discharge acid concentrations of 45 g/ℓ to 50 g/ℓ. None of the samples yielded nickel extractions below 95%. These results indicate the variability in nickel and cobalt extractions are minimal throughout the orebody and the robustness of HPAL to processing these ores.

Acid consumption for use in the process design and costings was assessed by two methods (Table 13-6):

- Method 1: Prediction of acid consumption by stoichiometry (using chemical equations and reaction extents) applied in process model and verification of adjustment between predicted and experimental acid consumption
- Method 2: Statistical determination of stoichiometric coefficients for acid consumption by assessing the experimental data.

Table 13-6 HPAL acid consumption equations

Item	R^2	HPAL acid consumption equations
Method 1	0.9955	16.03 (% NI) + 37.09 (% MG) + 31.8 (% AL) + 0.59 (% FE) + 88.2
Method 2	0.9965	16.3 (% Ni) + 40.9 (% Mg) + 23.5 (% Al) + 1.06 (% Fe) + 79.4

Source: GRD-Minproc, 2005

Three continuous pilot HPAL tests were completed during the PFS at Lakefield Oretest during late 2002 and early 2003. The residence time was nominally 60 minutes. When acid addition was controlled to target, metal extractions were typically greater than 95%. Data is presented in Table 13-7.

For the FFS testwork, composite samples from pilot plant beneficiation testwork were used. The composition of these samples was targeted to represent the first 10 years of operation, both from a geological and elemental grade point of view. Details from sample composition are presented in the SGS Lakefield Oretest report.

The target autoclave residence time of 60 minutes was selected from the PFS test data and was used for all the FFS testwork. Table 13-7 provides an average of the HPAL leaching data from runs T310, T320 and T330 respectively. All were very consistent and similar to the results obtained from the PFS testwork. Nickel extraction was typically greater than 96.5%. It is expected that similar extraction should be achievable in the full-scale industrial plant. Cobalt extraction was typically greater than 95.5%. Free acid levels of 55 g/ℓ to 60 g/ℓ were required to achieve the stated nickel and cobalt recoveries.

Table 13-7 PFS and FFS HPAL results summary

Run	Start date	End date	Nickel extraction (%)	Cobalt extraction (%)	Acid dose (kg/t)	Discharge free acid (g/ℓ)	Residence time (minutes)	Total iron (g/ℓ)	Ferrous (g/ℓ)
PFS Run 1	8/12/2002	14/12/2002	92.9	89.6	270.7	44.5	82.7	8.8	1.7
PFS Run 2	8/1/2003	14/1/2003	96.3	94.9	282.0	51.6	77.2	11.0	1.8
PFS Run 3	20/1/2003	26/1/2003	96.3	94.9	276.6	49.7	70.6	10.1	1.7
FFS T310	1/6/2004	14/6/2004	96.8	95.7	395.2	58.8	59.8	11.0	3.9
FFS T320	1/7/2004	16/7/2004	96.7	95.7	386	58.0	60.0	11.10	2.44
FFS T330	4/8/2004	18/8/2004	96.4	95.8	397	54.6	59.5	10.19	2.79

Source: GRD-Minproc, 2005

13.5 HZM high-pressure acid leach (HPAL) testwork

The previous PFS and FFS testwork programs completed in 2002 and 2004 respectively were focused on the HPAL/MHP/AR/MCP flowsheet option. This study has adopted an HPAL/MSP/POX/SX/crystallisation flowsheet. In late 2018 to early 2019, laboratory-scale testwork was carried out at SGS Lakefield (Ontario) on behalf of HZM to validate the suitability of the HPAL process and subsequent purification stages for processing Vermelho limonite to produce cobalt sulphate and nickel sulphate.

Approximately 157 kg (wet) material, comprising 16 samples, was shipped to the laboratories of SGS Lakefield (Ontario). These samples were considered representative of Vermelho limonite and were selected from material at site (from test material mined by the previous owner). Upon receipt and verification, the material was screened and then scrubbed in batches of about 25 kg with fresh water.

The as-received samples consisted of some lumpy material and fines. Figure 13-4 illustrates typical material during screening (Left hand side: +12 mesh/-0.5 inch; Right hand side: +0.5 inch/-1 inch).

Figure 13-4 Typical as-received Vermelho limonite samples

Source: HZM, 2019

The material was then screened with the final screen size of 35 mesh used (500 µm), with a rod mill used to grind the oversize so that 100% of the material was passing 35 mesh. The final P_{80} of the material proceeding to leaching was 147 µm.

A representative subsample of the sample was taken and submitted for elemental analysis. The analysis of the as-received ore based on the subsample is shown in Table 13-8.

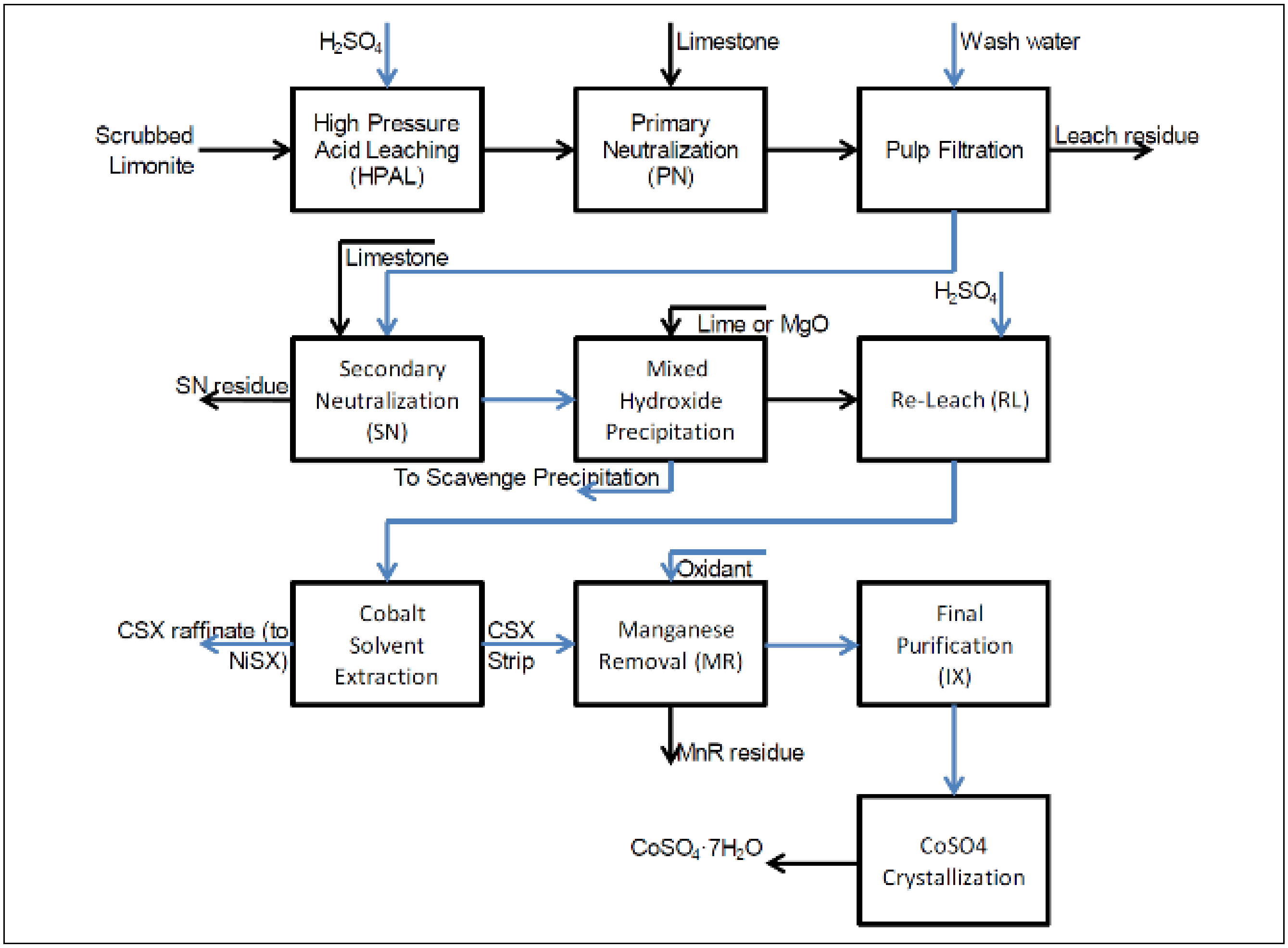
Table 13-8 Chemical analysis of Vermelho limonite sample

Item	Analysis, wt.% (dry basis), Fe/Ni and Ni/Co as a ratio										
	Ni	Co	SiO_2	Fe	MgO	Al_2O_3	CaO	MnO	Cr_2O_3	Fe/Ni	Ni/Co
Result	1.72	0.076	42.2	27.7	4.23	1.51	0.04	0.41	1.71	16.1	22.6

Source: HZM, 2019

A simplified process flow diagram that illustrates the methodology used in the testwork is shown in Figure 13-5. This testwork focused on producing battery-grade cobalt sulphate only (which was considered more challenging than Ni) with the intent to produce nickel sulphate solution ready for purification in subsequent work.

Figure 13-5 Simplified process flowsheet for the 2019 HPAL tests



Source: HZM, 2019

The testwork was intended to produce a leach solution for further testing. Consequently, pre-beneficiation of the ore and optimisation of leach conditions were not included. In earlier work on Vermelho ore by Vale, a pre-beneficiation step for ore upgrading by rejecting some of the magnesium silicates was included (Torres, 2003; Adams *et al.,* 2004).

The hydrometallurgical tests carried out are briefly summarised below and grouped into the four major test stages.

1) HPAL and neutralisation of solution:
   a) Leaching of the scrubbed and screened ore sample using sulphuric acid.
   b) Primary neutralisation of the leach solution using limestone.
   c) Filtration and secondary neutralisation using limestone.
2) Production of MHP and re-leaching:
   a) Using hydrated lime to produce MHP, primarily to separate nickel and cobalt from the iron. It is noted that the use of MHP was carried out for convenience, as in a commercial operation it is anticipated that MSP would be likely to be adopted as it recovers less impurities to the intermediate product.
   b) Re-leach of the MHP product using sulphuric acid.

3) Separation of cobalt and nickel and purification of the cobalt solution:
   a) Cobalt solvent extraction to separate nickel and cobalt.
   b) Purification of the cobalt solution by ion exchange.
   c) Cobalt sulphide precipitation and re-leach to further purify the cobalt solution.
   d) Final polishing of the solution for removal of the last traces of manganese.
4) Cobalt sulphate crystallisation[6]:
   a) Batch-wise crystallisation.

13.5.1 High-pressure acid leach and neutralisation of solution

To validate conditions for the subsequent larger scale HPAL leaching, two small scoping level leach tests were carried out in a 2 ℓ autoclave with approximately 400 g of sample at a total pressure of 585 psi. The outcome of these scoping tests combined with the Vale leach test conditions were used to set the conditions in the 20 ℓ test autoclave. This mode of leach testing was then carried out in using approximately 5.5 kg of un-beneficiated material from the sample per batch in the 20 ℓ autoclave. The typical leaching conditions used for these tests are presented in Table 13-9, below.

Table 13-9 Typical leaching conditions used for the testwork

Item	Unit	Value
Solids	%	35
Acid used	kg/t dry ore	350-400
Total pressure	psi	585*
Temperature	°C	250
Ni extraction	%	98
Co extraction	%	96

**The pressure was fixed at this level for the test.*

A total of 20 individual leach batches were carried out to process all the prepared sample. A typical analysis of the final pulp filtrate is presented in Table 13-10.

Table 13-10 Typical chemical analysis of the final pulp filtrate

Element	Analysis (mg/ℓ)
Ni	7,880
Co	345
Fe	7,660
Mg	10,600
Al	2,370
Cu	49.2
Zn	152
Mn	1,490
Cr	230

[6] This testwork focused on producing battery-grade cobalt sulphate only (which was considered more challenging than Ni) with the intent to produce nickel sulphate solution ready for purification in subsequent work.

Two limestone scoping tests (for the primary and secondary neutralisation) were successfully carried out to develop the conditions for solution neutralisation. The final pH was set at between 5 and 5.1. The primary and secondary neutralisation steps on the bulk solution were subsequently executed. In this test phase, several sub-products were generated which were **not** recycled as would occur in a continuous commercial operation; consequently, metal recovery, which is impacted by this situation, was **not** evaluated as part of this work.

13.5.2 Production of mixed hydroxide precipitate (MHP) and re-leaching

Hydrated lime $Ca(OH)_2$ was used to generate the MHP product containing mostly nickel and cobalt hydroxides, along with some calcium. The analysis of the MHP prior to re-leaching is presented in Table 13-11.

Table 13-11 Chemical analysis of the MHP

Element	Analysis (%)
Ni	18.3
Co	0.97
Mn	1.92
Mg	0.67
Ca	13.1
Cu	<0.01
Zn	0.12
Ni/Co	18.9

13.5.3 Separation of cobalt and nickel and purification of the cobalt solution

Solvent extraction

Test conditions were established using small-scale laboratory tests. In addition, Cytec Canada Inc. provided recommendations for test conditions (the modelling by Cytec recommended the use of a 10% Cyanex 272 reagent in Exxsol D80 with a phase ratio of 1:1; the scrub liquor was also defined). Loading and scrubbing for solvent extraction on the bulk solution was then carried out on a continuous basis with stripping undertaken batch-wise in laboratory apparatus that was assembled for the purpose.

The nickel sulphate solution was set aside for testing and purification at a later date. The cobalt sulphate solution was then further purified.

Ion exchange

Ion exchange was carried out as a two-stage ion exchange contacting sequence. A sulphide precipitation step was then carried out after the ion exchange step following initial tests using a synthetic solution to determine the best process conditions. Thermochemical computations showed that under the proposed sulphide precipitation conditions, cobalt sulphide would precipitate ahead of that for zinc, copper and manganese, thus affording a means of elemental separation and purification. A further manganese polishing stage using Caro's acid in three contact stages was also carried out.

13.5.4 Cobalt sulphate crystallisation

The near-pure cobalt sulphate solution was crystallised batch-wise under controlled conditions and a single harvest of cobalt sulphate crystals was gathered. This material at over 99.91% pure (Figure 13-6) is considered ready for a final purification stage for specific battery market applications.

The additional purification steps would potentially include, for example: re-dissolution and reaching a higher Co concentration (e.g. ~90 g/ℓ) followed by additional ion exchange, then another treatment with Caro's acid.

Figure 13-6 Image of the cobalt sulphate heptahydrate product

Source: HZM, 2019

13.6 Conclusion

The author considers the testwork described above is sufficient to support the selected process route and the beneficiation assumptions, and the metallurgical recoveries used in the economic analysis.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

During February 2018, the author reviewed the historical MREs and concluded that the “FFS_25_2_m” MRE that was audited by Snowden in 2005, is appropriate for adoption as a current MRE and concluded that the reporting conforms to the requirements of the CIM Definition Standards (2014).

Within the mining licence, at a cut-off grade of 0.7% Ni, a total of 140.8 Mt at a grade of 1.05% Ni and 0.05% Co is defined as a Measured Mineral Resource; and a total of 5.0 Mt at a grade of 0.99% Ni and 0.06% Co is defined as an Indicated Mineral Resource. This gives a combined tonnage of 145.7 Mt at a grade of 1.05% Ni and 0.05% Co for Measured and Indicated Mineral Resources. A further 3.1 Mt at a grade of 0.96% Ni and 0.04% Co is defined as an Inferred Mineral Resource at a cut-off grade of 0.7% Ni.

The author is not aware of any issues that materially affect the Mineral Resources in a detrimental sense.

14.2 Method

The “FFS_25_2_m” estimates were prepared in the following steps by Vale, reviewed by Snowden in 2005 and again reviewed by the author in 2018:

- 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 in unfolded space
- Model validation
- Calculation of dry density
- Classification of estimates with respect to the CIM Definition Standards (2014) for Mineral Resources (by the author)
- Resource tabulation and resource reporting (by the author).

Description of the MRE process is described in Snowden (2005) and reviewed extracts are provided below.

14.3 Data preparation

This is discussed in Sections 10, 11 and 12.

14.4 Geological interpretation

14.4.1 Topography

Drillhole collar locations were checked against the topographic surface provided by Vale and no errors were found.

It was noted that the topographical surface extended below the interpreted base of cover and base of the High ("Alto") Al_2O_3 surfaces. Vale confirmed that the topographical surface was correct and that the other surfaces were adjusted to this surface during the block modelling process. A minimum surface was generated to correct for surface overlaps and all surfaces and solids were adjusted to be below the topographical surface. The percentage rock field was checked against the topographical surface and no errors were found. Based on this percentage rock field a percentage ore and a percentage waste were then assigned to the block model. These were checked to ensure that the percentage ore and percentage waste equalled the percentage rock below topography.

Resource reporting was checked to determine that the percentage rock field was implemented during resource reporting and it was confirmed that the final resource reports for V1 and V2 have been reported below topography.

14.4.2 Mineralised domains

The resource is contained within limonite and saprolite horizons and the base of the resource is defined by the extent of the +0.25% Ni envelope. The base of the saprolite with Low Ni (with 0.25% to 0.5% Ni) has been interpreted as a relatively smooth surface. It is understood the bedrock surface between the saprolite and the bedrock (the weathering front) is not as smooth as has been interpreted from the available data by Vale, and that it is generally extremely irregular. However, as the base of the resource model is generally above the bedrock surface, this is not regarded as being an issue for the resource model.

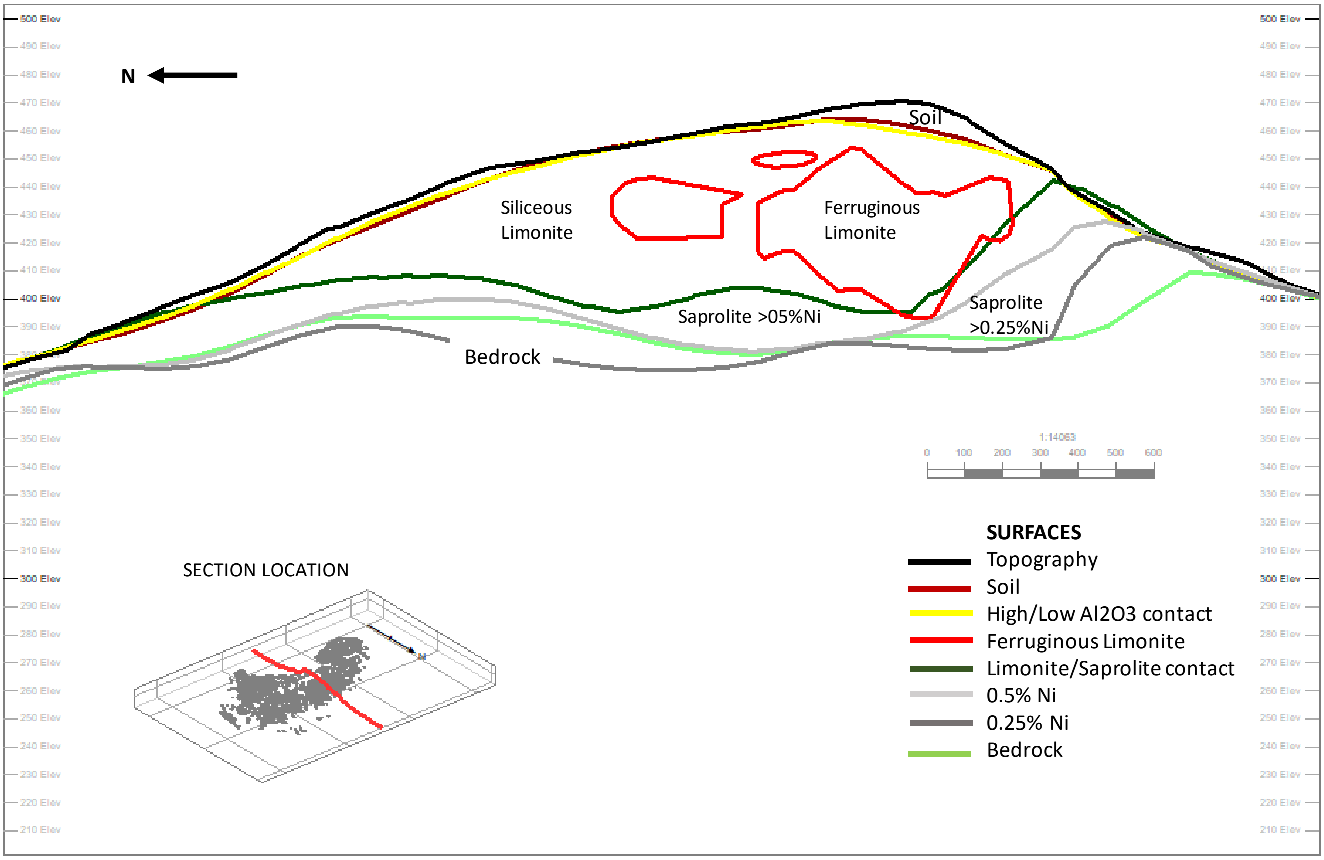
The resource models were domained based on a combination of weathering, lithology and chemistry. The mineralised horizon has been subdivided into five domains for resource modelling at V2. These domains are as follows: Soil/Ferricrete, siliceous limonite, High Ni saprolite, Low Ni saprolite and ferruginous limonite (Figure 14-1). Majority of the resource is contained within the siliceous limonite and saprolite horizons.
In addition to these domains, a sixth domain which comprises gabbro dykes has been used for the resource modelling process. At V1, the gabbro dykes have a steep, and often vertical orientation, with some interpreted near horizontal lenses branching off the main dykes (*Source: HZM, 2019*

Figure 14-2).

The digitised outlines used to represent these interpretations in Gemcom were reviewed on section. The 3D solid models and surfaces that were subsequently developed from these outlines were also reviewed and were used to control the resource estimation process at V1 and V2.

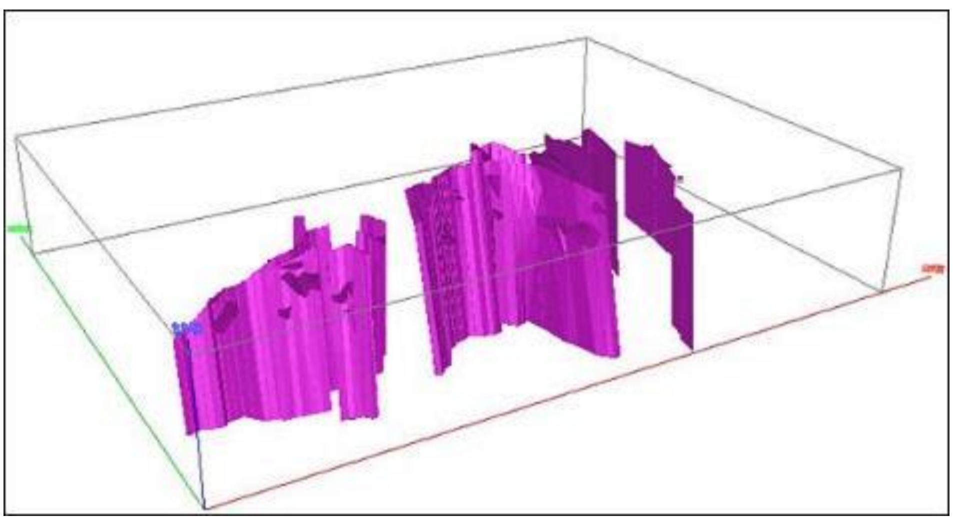
The author is satisfied that all stages of the geological interpretation have been carried out in a very thorough and consistent manner and considers that the finalised 3D domains provide a good representation of the geological interpretation.

Figure 14-1 Typical schematic section showing geological domains used for resource models



Source: HZM, 2019

Figure 14-2 Extent of gabbro dykes at V1



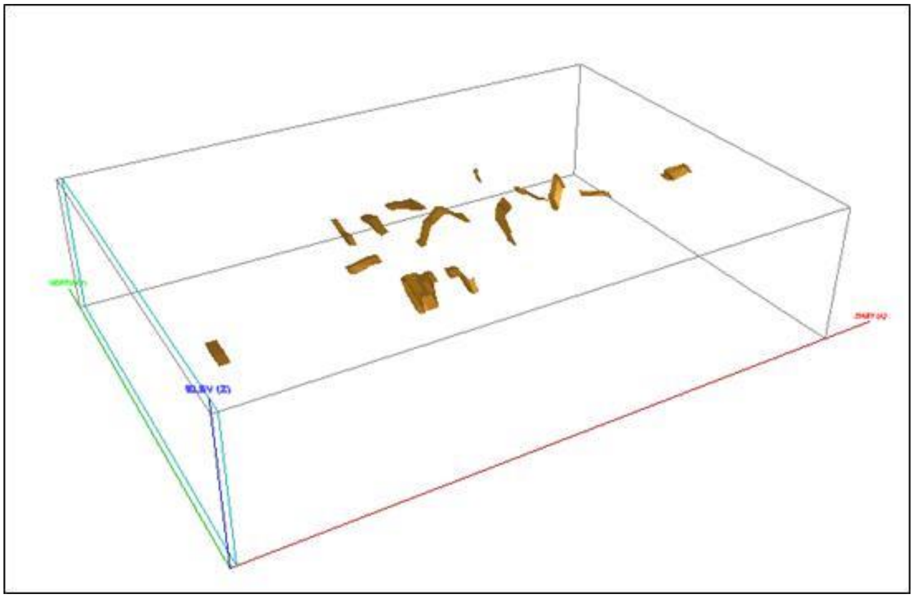
Source: HZM, 2019

14.4.3 Dilution

Dilution has been incorporated into the resource model as internal waste (as illustrated in Figure 14-3 and Figure 14-4) and at the base of the resource models. The example section from V2 (Figure 14-4) illustrates the block model coloured by the waste percentage that has been incorporated into the blocks, which shows the majority of the waste has been added at the base of the Waste saprolite surface.

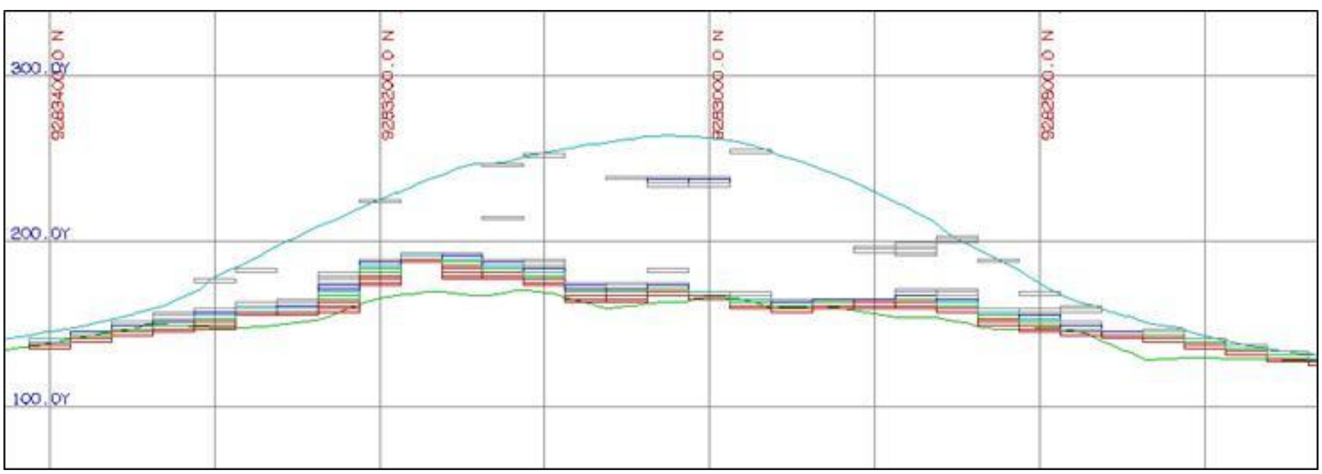
Diluted models were generated for V1 and V2, based on the assumption that the entire 25 m(E) x 25 m(N) x 2 m(RL) block would be mined, and incorporated a background grade for the waste component of each block.

Figure 14-3 Internal waste at V2



Source: HZM, 2019

Figure 14-4 V2 along block model 621937.5 mE (column 30) – percentage waste incorporated into blocks



Note: 0.01–20% grey, 20–40% blue, 40–60% light blue, 60–80% green, 80–100% red

Source: HZM, 2019

14.4.4 Coding of data and block model

Coding of the input data was checked by confirming the location of the coded data with respect to the surfaces and solid models. The locations of coded blocks were also checked against these surfaces and solids. No incorrectly coded data or blocks were apparent.

14.5 Data analysis

14.5.1 Statistical analysis

The domain coding used by Vale identified five weathering domains: cover, saprolite, SapFe, bedrock and boulders (for updated nomenclature developed by HZM refer to Figure 7-5), three chemical domains (High or "Alto" Al_2O_3, High or "Alto" MgO and Low or "Baixo" MgO), two domains based on nickel grade (High >0.5% Ni and Low 0.25% to 0.5% Ni) and four geological domains (gabbro, pyroxenite, serpentinite and boulders). Combinations of the weathering, chemical, Ni and geological domains have resulted in the definition of 43 sub-domains at V2 and 61 sub-domains at V1. Detailed statistical analysis, undertaken by Vale and presented to Snowden in the form of graphs and tables, was used to combine these sub-domains into the final five mineralised domains for grade estimation at V2 and the final six mineralised domains for grade estimation at V1.

There is some degree of population mixing present in some of the domain/element distributions, but the grade distributions within each of the domains are generally well defined and are suitable for grade estimation. The coefficients of variation are generally less than one and where they are higher this has been addressed by grade capping.

14.5.2 Variogram analysis and kriging parameters

Detailed variography was undertaken by Vale in an unfolded coordinate system for each variable within the domains defined for grade estimation. Two reports containing variogram maps, directional variogram plots and interpreted models and tables of interpreted parameters were supplied to Snowden. Following the review of the V2 variography results, revisions were made to the interpreted orientation of maximum continuity in the ferruginous limonite or "SapFe" domain at V2.

14.5.3 Bulk density data

The bulk density data available for the Vermelho resource estimate comprised 727 measurements for V1 and 811 for V2. Detailed analysis of the density data in relation to data collection methodology, moisture content, depth and rock domain (including weathering) was undertaken by Vale. Apparent discrepancies in the data were investigated by Vale and were explained by the selection of the method being dependent on the nature of rock being measured. Vale determined that all the data was valid. The strategy developed for ore and waste density estimation into the model was based on a combination of estimation techniques that were controlled by the rock domain and the amount of data available. Estimation methodologies applied include ordinary kriging, nearest neighbour from point data, attributing the mean from the adjacent blocks, and attributing the de-clustered mean from the adjacent blocks. The estimation was undertaken in real coordinate space (i.e. not unfolded). Component ore density estimates for each "rock type" folder was volume weighted to develop the integrated undiluted and diluted ore density models for V1 and V2.

Snowden considers that the ore and waste density models are of a good standard and are suitable for a feasibility-level study.

14.6 Unfolding

Snowden endorsed the use of unfolding to improve grade connectivity in the variography analysis and the estimation process. Data was unfolded to the base of the High Al_2O_3 domain to control variography analysis and block grade estimation of all grade variables into the High Al_2O_3 domain and Al_2O_3 into the Low MgO domain. The ferruginous limonite, Low MgO, High Ni Low MgO and Low Ni High MgO data were unfolded to the base of the Low MgO domains to control variography analysis and block grade estimation of all variables into these domains, except the Al_2O_3 in the Low MgO domain as mentioned in Section 14.5.1.

Gemcom's unfolding transformation can generate artefacts such as well-positioned blocks close to data remaining uninformed. To overcome this problem, Vale used the estimated grades from adjacent blocks to determine the grade of the uninformed blocks using a nearest neighbour technique; blocks that were affected in this way account for less than 1% of the total blocks. This was also addressed during the classification, whereby blocks that were not estimated during kriging due to unfolding problems were given an Inferred classification. As mentioned in Section 14.9, the classification was revised where the blocks were intersected by a drillhole, and after confirmation that the block grades reflected the drillhole grades, these were given an Indicated classification.

14.7 Grade estimation

Ordinary kriging was applied to generate block estimates for the Ni, Co, MgO, Al_2O_3, Fe_2O_3, MnO, SiO_2, Cr, Cu and Zn. In addition to grade capping, a restricted search was applied that restricted the influence of grades above the 97.5th percentile to 25 m(E) x 25 m(N) x 2 m(RL).

14.8 Block model validation

Analysis of the block models showed that there were no uninformed blocks and that, with the exception of a few problem density values which were addressed, each data field contained permissible entries – i.e. all values were within allowable ranges and there were no foreign codes evident.

3D visualisation of the drillhole data and the block model grades indicates good visual validation of the block model grades. Trend plots were provided by Vale for all grade variable and domain combinations. These illustrated the mean block grades, along with the mean grade of the data and the numbers of input data, by incremental northing, easting and elevation. The validation plots all indicate that there is a good correlation between the input grades and the block grades.

14.9 Classification

Vale applied sample spacing criteria for resource classification of the V1 and V2 models. The criteria were developed from the nickel variography and the number of search points within the search ellipse, the number of points used for block grade estimation, the anisotropic distance to the nearest point, and the kriging run number were used for classification. Snowden checked the implementation of the classification criteria and, following discussion with Vale, the classification of some blocks was amended by Vale. Blocks that had been classified as Inferred and that were intersected by a drillhole, were upgraded to an Indicated classification, after confirmation that the block grades reflected the drillhole grades. For some blocks, the criteria for the number of points (used for estimation and/or within the search ellipse) was relaxed in areas where the domains were thin (which limited the sample availability) but had been sufficiently drilled to warrant a Measured classification.

The author considers that this is an appropriate method to classify a global resource estimate based on nickel grade in accordance with CIM Definition Standards (2014).

14.10 Integration of component models

Final diluted and undiluted models were developed by integration of the five component rock folders for V2 and six component rock folders for V1 for all the grade and density variables. The variables were volume weighted by the percentage ore present in each rock type to obtain the final block grade and ore density for the undiluted models. As mentioned in Section 0, a background grade was assigned to the waste component of each rock to derive a diluted grade for the integrated models.

The Gemcom scripts for the volume weighting of the variable were checked and no errors were found. However, it appears that there may be some precision problems with low grade variables in Gemcom and errors were noted in some of the final undiluted Fe_2O_3 grades, for the V2 model, and final undiluted Cu grades, for the V1 model. These were corrected by Vale as part of the review process.

14.11 Resource reporting

The classified resource, based on the undiluted block models, reported by Vale for V1 and V2 were checked by Snowden and the author. The block models are reported using Gemcom v 6.2 (Table 14-1).

Table 14-1 V1 + V2 – combined classified Resource report for Vermelho by Ni cut-offs within the mining licence

Cut-off (Ni %)	Tonnage (Mt)	Ni (%)	Ni metal (kt)	Co (%)	Co metal (kt)	Fe_2O_3 (%)	MgO (%)	SiO_2 (%)
Measured								
0.6	173.1	0.97	1,688	0.05	89.0	31.7	10.2	42.0
0.7	**140.8**	**1.05**	**1,477**	**0.05**	**74.6**	**31.1**	**11.3**	**41.0**
0.8	111.5	1.13	1,258	0.05	60.1	30.1	12.3	40.6
0.9	84.6	1.22	1,030	0.05	45.9	28.8	13.4	40.5
1.0	63.2	1.31	827	0.05	34.4	27.4	14.4	40.6
Indicated								
0.6	6.8	0.90	61	0.05	3.6	26.0	9.1	49.2
0.7	**5.0**	**0.99**	**49**	**0.06**	**2.8**	**26.3**	**8.6**	**49.0**
0.8	3.7	1.08	40	0.06	2.2	26.3	8.6	48.7
0.9	2.6	1.17	31	0.06	1.6	26.1	8.7	48.3
1.0	1.9	1.27	24	0.06	1.1	25.8	9.0	47.7
Measured+Indicated								
0.6	179.9	0.97	1,749	0.05	92.6	31.5	10.2	42.3
0.7	**145.7**	**1.05**	**1,526**	**0.05**	**77.3**	**30.9**	**11.2**	**41.3**
0.8	115.2	1.13	1,298	0.05	62.3	30.0	12.2	40.8
0.9	87.2	1.22	1,060	0.05	47.5	28.7	13.3	40.7
1.0	65.1	1.31	851	0.05	35.5	27.4	14.2	40.8
Inferred								
0.6	4.9	0.84	41	0.04	1.9	22.3	17.3	42.1
0.7	**3.1**	**0.96**	**29**	**0.04**	**1.4**	**24.0**	**15.5**	**42.2**
0.8	2.1	1.06	22	0.05	1.0	25.1	14.0	42.5
0.9	1.4	1.16	17	0.05	0.8	25.6	13.2	42.4
1.0	1.0	1.26	12	0.06	0.6	26.5	12.8	41.5

Notes:

- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive subtotals, totals and weighted averages. Such* *calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Snowden does not consider them to be material.*

- *Mineral Resources are reported inclusive of Mineral Reserves.*
- *The reporting standard adopted for the reporting of the Mineral Resource estimate uses the terminology, definitions and guidelines given in the CIM Definition Standards (2014) on Mineral Resources and Mineral Reserves as required by NI 43-101.*
- *Mineral Resources are reported on 100% basis for all Project areas.*
- *Snowden completed a site inspection of the deposit by Mr Andy Ross (FAusIMM), an appropriate "independent qualified person" as such term is defined in NI 43-101.*
- *kt = thousand tonnes (metric).*

15 MINERAL RESERVE ESTIMATES

Mineral Reserves were prepared for the Project as part of the PFS, using the CIM Definition Standards (2014).

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 in conjunction "estimation of Mineral Resource and Mineral Reserve best practice guidelines" as provided by the CIM. No Mineral Reserves have been estimated using Inferred Mineral Resources.

The Reserves use the assumptions, designs and parameters defined predominantly in Section 16 and from other relevant sections of this report, applied as modifying factors.

15.1 Summary

The estimation of Mineral Reserves used the Measured and Indicated Mineral Resources for the Project as reported in Section 14 of this report.

All economic Measured and Indicated Resources within the pit designs were classified as Probable Reserves. A summary of the Mineral Reserves is provided in Table 15-1.

Table 15-1 Vermelho Mineral Reserve estimates, as at October 2019

Value	Probable
Ore (Mt)	141.3
Ni (%)	0.91
Co (%)	0.052
Fe (%)	23.1
Mg (%)	3.81
Al (%)	0.79

Notes:

- *Cut-off varies by resource model block depending on individual block geochemistry, however, as a guide the cut-off is approximately 0.5% Ni.*
- *Dilution was modelled as part of re-blocking, ore losses applied are 2%.*
- *Snowden completed a site inspection by Mr Anthony Finch P.Eng. MAusIMM (CP Min.), an appropriate "independent qualified person" as such term is defined in NI 43-101.*

15.2 Key assumptions, parameters and methods

15.2.1 Methodology

The mine plan followed a process of pit optimisation, design and scheduling.

Pit optimisations were completed in Whittle Four-X™ 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 block and the maximum allowable slope angles. Nested pit shells produced by the pit optimisation were used in the selection of the "optimum" pit shell and for guiding the location of pit stages.

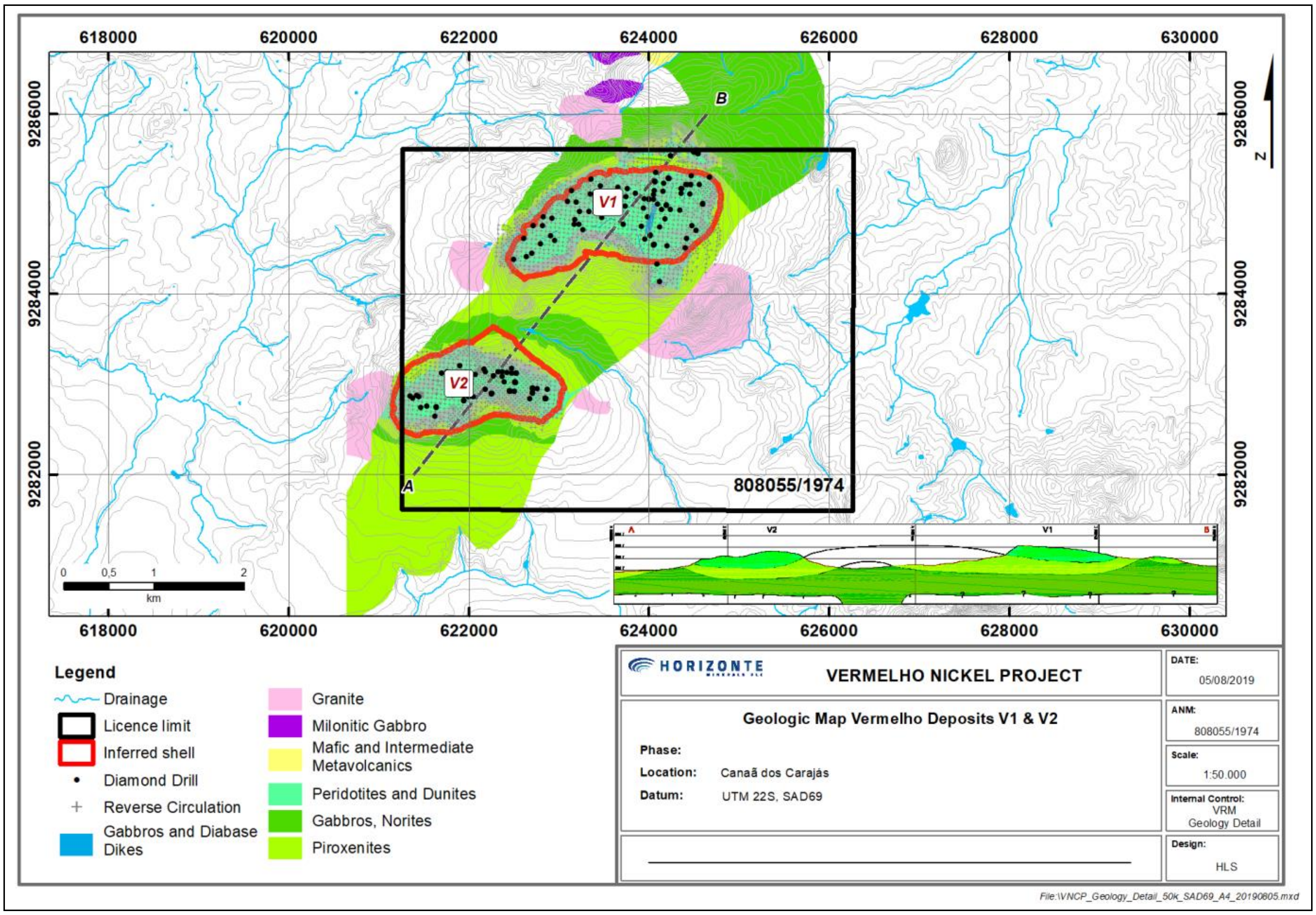
Using the selected pit shells, pit designs for the final pit limits and stages were developed in MineSight®. The pit designs were used to derive volumes for waste and for final ore tonnages used in scheduling. Feedback from all relevant stakeholders was used to determine a waste disposal concept for each deposit, including both ex-pit and in-pit options. After calculating the volumes of waste, waste dumps were designed to contain this material and minimise required haulage distances.

15.2.2 Mining model

The Gemcom resource model created by Vale was reviewed by Snowden in late 2004/05. Snowden converted this model into Datamine format. Only Measured and Indicated Resources within the mining licence area (Figure 15-1) were considered for mine planning. Grade tonnage data is listed in Table 15-2.

The resource model includes dilution in the sense that grades were averaged to the size of model blocks (i.e. 25 m x 25 m x 2 m).

Figure 15-1 VMH mining licence area



Source: HZM, 2019

Table 15-2 Grade tonnage data (Measured and Indicated Resources only, within mining licence)

Ni % cut-off	Mass (Mt)	Ni (%)	Ni metal (kt)	Co (%)	Co metal (kt)	Fe_2O_3 (%)	MgO (%)	SiO_2 (%)
0.1	287.7	0.78	2,246	0.04	120.6	31.0	8.5	45.7
0.2	287.7	0.78	2,246	0.04	120.6	31.0	8.5	45.7
0.3	285.8	0.78	2,240	0.04	120.3	31.1	8.4	45.7
0.4	260.2	0.83	2,148	0.04	116.5	31.5	8.4	45.2
0.5	219.4	0.90	1,965	0.05	106.8	31.6	9.1	43.8
0.6	179.9	0.97	1,749	0.05	92.6	31.5	10.2	42.3
0.7	145.7	1.05	1,526	0.05	77.3	30.9	11.2	41.3
0.8	115.2	1.13	1,298	0.05	62.3	30.0	12.2	40.8
0.9	87.2	1.22	1,060	0.05	47.5	28.7	13.3	40.7
1.0	65.1	1.31	851	0.05	35.5	27.4	14.2	40.8
1.1	93.5	1.40	1,309	0.05	50.3	25.9	15.4	40.8
1.2	68.1	1.49	1,018	0.05	36.4	25.0	16.0	40.8
1.3	50.1	1.58	794	0.05	26.5	24.3	16.6	40.7
1.4	35.7	1.68	599	0.05	18.8	23.9	17.0	40.5
1.5	25.4	1.77	451	0.05	13.3	23.4	17.5	40.3
1.6	18.1	1.86	337	0.05	9.5	23.3	17.7	40.0
1.7	12.8	1.95	251	0.05	6.8	23.2	18.0	39.6

Beneficiation of the ore was identified by Vale as a key opportunity for the Project. Based on metallurgical testing (Section 13), Vale developed equations that model the behaviour of the ore through the proposed beneficiation plant. These equations were reviewed as part of this Technical Report (Section 13) and are still deemed appropriate for use in mine planning. The beneficiation equations developed by Vale (Table 15-3) were applied to resource model in such a way that for each resource model cell, post-beneficiation values were calculated for each of the items in Table 15-3, based on the raw model values. These post-beneficiation values were used in pit optimisation and subsequent scheduling.

Table 15-3 Beneficiation equations (to calculate factor for the change the element after beneficiation)

Element	Factor equation
Mass recovery (%)	85.58741 – 1.592142*Si
Ni	1.083 – 0.01828*Si + 0.001786*Si^2
Co	0.9975 – 0.01913*Si + 0.001663*Si^2
Mg	1.018 – 0.01362*Si + 0.001238*Si^2
Fe	1.092 – 0.02168*Si + 0.001915*Si^2
Al	1.147 – 0.05443*Si + 0.002883*Si^2
Si	1.029 – 0.03714*Si + 0.000621*Si^2
Cr	0.8939 – 0.03727*Si + 0.001779*Si^2
Zn	1.171 – 0.04988*Si + 0.002740*Si^2
Cu	1.237 + 0.2348*Si – 0.004481*Si^2
Mn	1.092 – 0.03133*Si + 0.002034*Si^2

Because acid consumption is a key driver of Project economics, the calculated acid consumption was added to the resource model according to the HPAL feed chemistry using the equation below. The acid consumption was calculated on the beneficiated material (i.e. the HPAL feed) This equation was developed as part of the process engineering work completed by Simulus, and is discussed in Section 13:

- Acid consumption (kg/t HPAL feed) = 16.03 x Ni + 37.09 x Mg + 36.91 x Al + 0.5 x Fe + 88

The results of the application of the beneficiation equations and the acid consumption equation to the resource model are summarised in Table 15-4 (note that these equations apply regardless of rock type).

Table 15-4 Resource by domain with beneficiation outputs (Measured and Indicated Resources within mining licence)*

Rock type	ROM							Concentrate								Acid (kg/t concentrate)
	Mass (Mt)	Ni (%)	Co (%)	Si (%)	Mg (%)	Fe (%)	Al (%)	Mass (Mt)	Ni (%)	Co (%)	Si (%)	Mg (%)	Fe (%)	Al (%)	Recovery (%)	
SLIM	79.5	0.85	0.06	23.4	1.96	23.2	0.79	38.5	1.38	0.09	11.5	2.69	37.5	1.12	48%	309
SAP	74.3	1.13	0.04	19.5	12.33	13.9	0.60	40.6	1.59	0.04	10.5	15.08	19.4	0.71	55%	749
FLIM	24.0	0.88	0.06	8.7	0.67	43.6	1.40	17.2	0.95	0.06	6.0	0.67	47.3	1.33	72%	240
GAB	0.6	1.15	0.05	19.1	5.66	17.2	4.43	0.4	1.60	0.06	10.4	6.85	23.7	5.16	55%	588
Total	**178.4**	**0.97**	**0.05**	**19.8**	**6.11**	**22.0**	**0.81**	**96.6**	**1.39**	**0.07**	**10.1**	**7.55**	**31.6**	**1.00**	**54%**	**483**

**Nickel grades at greater than 0.6%; and reported Resources are all within the mining lease.*

15.2.3 Pit optimisation

Parameters and modifying factors

Pit optimisations were completed based on parameters derived from earlier studies and updated to reflect current conditions. These parameters are detailed below:

- Only Measured and Indicated Resources were considered for pit optimisation. Inferred Resources within the final pits were reported, but not included as potential plant feed.
- All deposit starting surfaces were the topographies used to generate the resource models. It is important to note that this surface differs in areas from the overall site topography. Snowden recommends reconciling the two surfaces prior to the next study.
- Each deposit was restricted to the mining licence boundary. An offset of 10 m was applied to allow access around the pit.
- A 30° overall wall angle was applied for pit optimisation after considering the geotechnical requirements and the need for ramps.
- A 2% mining loss was applied to the model.
- An HPAL process rate of 2 Mt/a was applied.
- An HPAL (and downstream) recovery of 92% for nickel and 90% for cobalt was applied.
- A mining cost of US$3.5/t rock was applied to all blocks based on experience with similar projects. A US$5.81/t ore incremental ore cost was added which reflects selectivity and grade control costs involved in ore mining.
- A beneficiation cost of US$4.5/t ore was applied.
- An administration cost of US$11/t HPAL feed was applied.
- The process cost was derived by Simulus (Section 21) and has the following components:
  - Acid cost – US$25.87/t acid required
  - Fixed cost – US$39.7/t HPAL feed
  - Limestone cost – ((16.03*Ni%+(31.8*Al%+0.59*Fe%)*3/2+88.2)*100/98*1.05)/1000*43 US$/t HPAL feed
  - Hydrogen sulphide cost – (Ni%/100*34.1/58.7+Co%/100*34.1/58.9)*1.05*155) US$/t HPAL feed
  - Caustic soda cost – (((0.1*Ni%/100+1.2*Co%/100)*40/58.7)*585) US$/t HPAL feed.
- A royalty of 2% was applied.
- A nickel price of US$14,000/t Ni and cobalt price of US$32,000/t Co was applied. Payability of 100% was assumed.
- The marginal cut-off grade (based on the above parameters) varies on a block-by-block basis, depending on its input geochemistry. As a guide, the cut-off is approximately 0.5% Ni.

Optimisation results

Pit shells were selected on the basis of meeting an HZM defined target of a 35-year mine life. This occurs at the 0.6 revenue factor (RF) pit shell (shell 16). A summary of the selected pit shells is shown in Table 15-5. The results indicate that a significantly larger project is possible (either supporting a higher production rate or extending the mine life), with the RF1 pit being almost double the size of the selected shell.

Table 15-5 Pit optimisation – selected shells

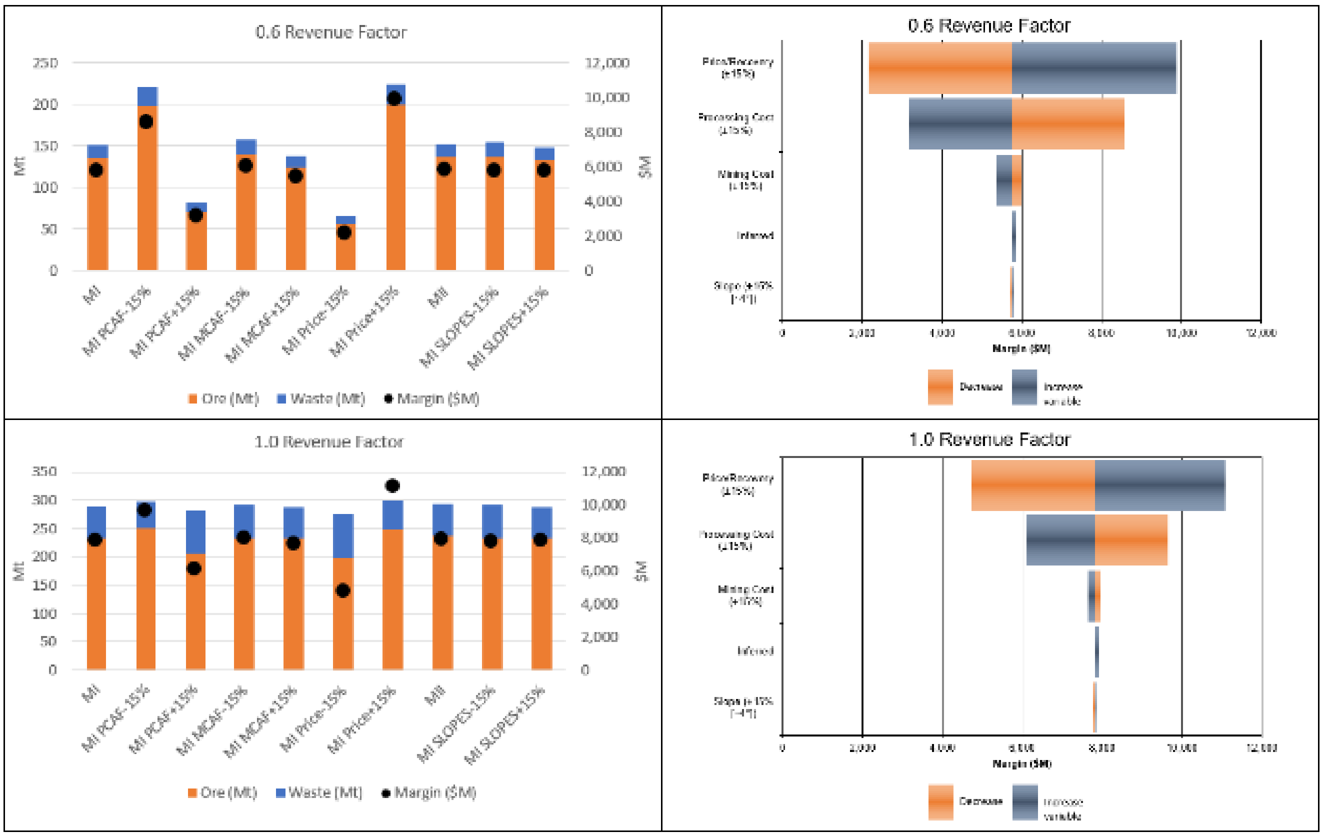
Pit shell	RF	ROM									Concentrate						Plant		
		Pit size (Mt)	Strip ratio (w:o)	Waste (Mt)	Ore (Mt)	Ni (%)	Co (%)	Mg (%)	Al (%)	Fe (%)	Mass (Mt)	Ni (%)	Co (%)	Mg (%)	Al (%)	Fe (%)	Acid (kg/t)	Ni recovery (kt)	Co recovery (kt)
11	0.5	54.5	0.08	4.4	50.0	1.04	0.06	3.74	0.83	22.1	25.3	1.61	0.08	4.95	1.12	34.3	353	373	18.6
16	**0.6**	**147.9**	**0.10**	**15.4**	**132.5**	**0.92**	**0.05**	**3.49**	**0.80**	**23.4**	**67.7**	**1.40**	**0.07**	**4.56**	**1.07**	**35.9**	**335**	**874**	**44.6**
23	0.8	266.3	0.17	44.6	221.7	0.84	0.05	3.87	0.76	22.6	112.1	1.30	0.07	5.11	1.04	35.0	352	1,341	66.5
27	1.0	288.3	0.19	54.2	234.1	0.84	0.05	4.26	0.76	22.2	118.8	1.29	0.06	5.61	1.03	34.3	370	1,406	68.8
37	2.0	320.3	0.26	83.0	237.2	0.84	0.05	4.37	0.76	22.1	120.5	1.28	0.06	5.75	1.03	34.0	375	1,423	69.3

Sensitivity results

Pit optimisations were completed on a number of scenarios focusing on ±15% adjustments on commodity prices, processing cost, mining cost and slope angles. Additionally, a pit optimisation including Inferred Resources was completed (MII). These were completed at the selected RF (0.6) and RF 1.0 (Figure 15-2).

The optimisation is highly sensitive to price/recovery and process cost. Inclusion of Inferred Resources or improvements in mining costs or slope angles do not have a large relative impact on the result.

Figure 15-2 Pit optimisation sensitivity results



Source: Snowden, 2019

15.2.4 Pit design

Geotechnical guidelines used for pit design were based on Table 15-6 from Golder (2004).

Table 15-6 Geotechnical guidelines for pit design (Golder, 2004)

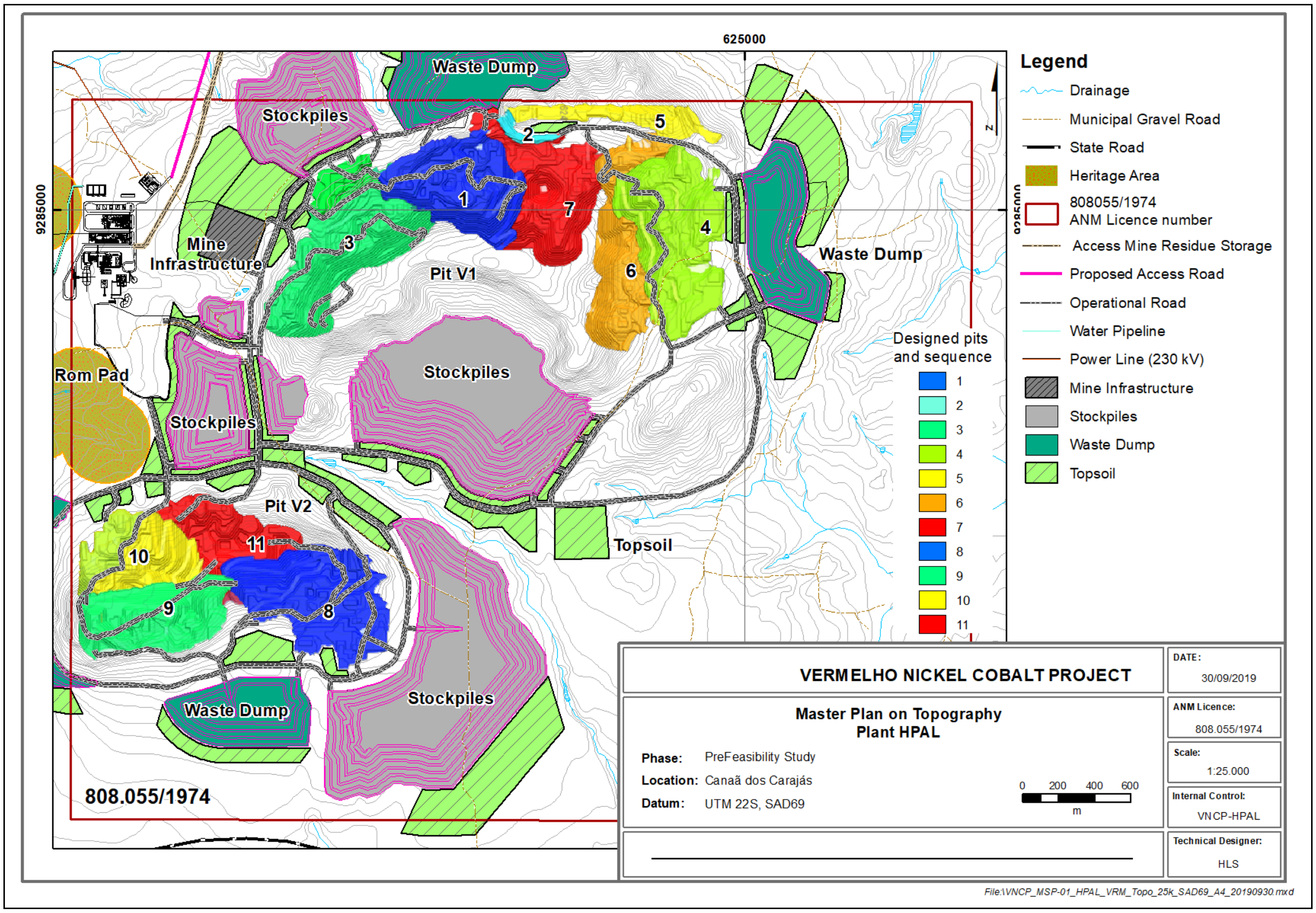
Height of wall (m)	Face height (m)	Berm width (m)	Face angle (°)	Overall angle (°)
<40	10	5	58	42
40<H<=60	10	5	53	39
60<H<=80	10	5	48	36
80<H<=100	10	5	43	33

Due to the difference in selected bench height (8 m) and to simplify the design process, all walls used face angles of 53° and berm widths of 6.1 m to achieve an overall angle of 33°, regardless of the wall height.

All roads and ramps were designed to be 20 m wide with a maximum angle of 10% for internal pit. The minimum width for all stages was 80 m with a minimum mining width of 20 m. Pit floors were designed flat but will need to be inclined in execution designs, to aid drainage in the wet season, once grade control and final limits are determined.

Figure 15-3 shows the pit split into 11 stages, seven stages in V1 and four stages in V2. The stages were ordered to follow the progression of the nested pit shells from the optimisation, whilst considering practical aspects of access in complex terrain.

Figure 15-3 Ultimate pits with stages



Source: HZM, 2019

A comparison of the design inventory to the pit optimisation shell (Table 15-7) shows that the design takes additional ore and waste when compared to the pit shell. This is primarily due to the design's smoothing walls and pit floors. As a low RF pit shell was selected, the additional material included is economic and improves the overall cash flow of the pit. This reconciliation is considered acceptable for this level of study.

Table 15-7 Pit design reconciliation to optimisation shell

Item	Pit shell	Design	Difference
Pit size (Mt)	147.9	161.6	+9%
Waste (Mt)	15.4	20.2	+31%
Ore (Mt)	132.5	141.3	+7%
Ni (%)	0.92	0.91	-1%
Cash flow (US$ M)	5,749	5,915	+3%

15.3 Disclosure

The Mineral Reserves were estimated under the supervision of Mr Anthony Finch who is a Qualified Person as defined in NI 43-101, employed by Snowden. In addition, Mr Simon Walsh employed by Simulus acts as the Qualified Person for metallurgical testwork and metallurgical parameters including plant recoveries and operating and capital costs. The abovementioned Qualified Persons relied on other experts and HZM for other items such as permitting, in-country social engagement, marketing, and metal pricing.

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.

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

Permitting is not expected to be a material risk to the Project as there have been no indications 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 that hinder the estimation of a Mineral Reserve. The infrastructure is either existing or of a relatively standard type to install during construction of the Project.

16 MINING METHODS

The mining study was completed by Snowden under the supervision of Mr Anthony Finch, an employee of Snowden. Mr Finch is the Qualified Person for Mineral Reserves and overall Qualified Person for the study; and he has attended site visits to the Project on four occasions between July 2018 and September 2019.

The mine design and accompanying schedules, detailed herein, are based upon on a planned open pit nickel laterite mining operation that mines a 141.3 Mt Mineral Reserve over a 39-year mine life.

In this section, all masses are reported as dry metric tonnes unless otherwise indicated and all currency is in US$ unless otherwise indicated.

16.1 Vermelho deposit

16.1.1 Size, shape and attitude

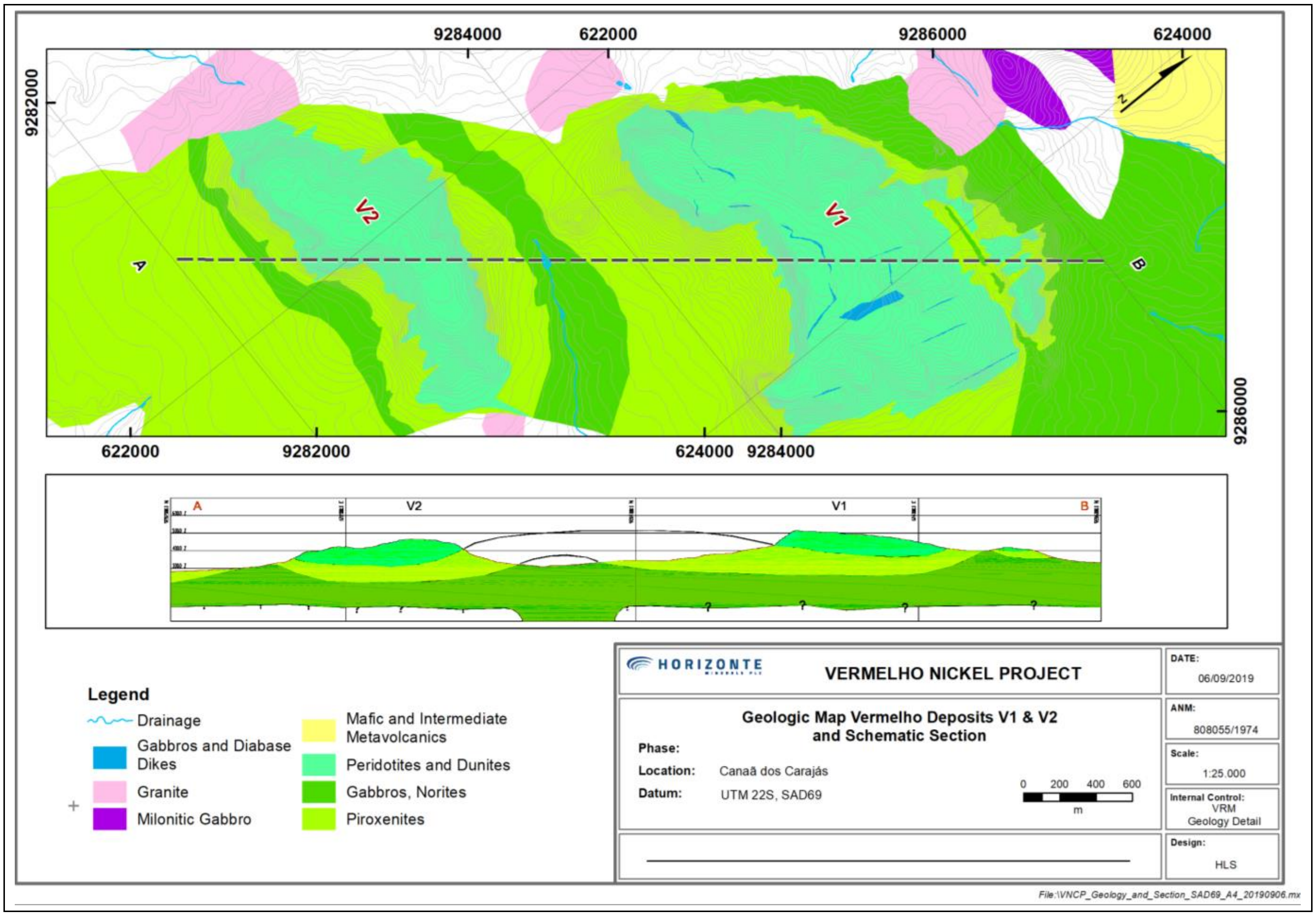
The Project comprises two deeply weathered ultramafic bodies (V1 and V2) that are interpreted (Section 14.4) to have had an extensive history of tropical weathering which has produced a thick lateritic profile of nickel enriched limonites and saprolites. The deposits were formed by the supergene enrichment of nickel and cobalt by the weathering of ultramafic rocks.

The deposits are flat lying and form topographical highs, with the V2 deposit being located approximately 1.5 km to the southwest of V1 (Figure 15-1). V1 is approximately 2.4 km east-west and ranges from 700 m to 1.6 km north-south. V2 is approximately 1.9 km east-west and ranges from 600 m to 900 m north-south. The V1 deposit has an average thickness of 53 m and a maximum thickness of 146 m, and the V2 deposit has an average thickness of 56 m and a maximum thickness of 115 m.

16.1.2 Rock types

The mineralisation at Vermelho is contained within a serpentinite that has been intensely weathered to laterites and is underlain by pyroxenite and gabbro (Figure 16-1). The V1 deposit has been intruded by near vertical gabbro dykes. The Vermelho resource has been geologically modelled according to variations in lithology and in the chemical and physical properties, which are controlled by the interaction of the weathering process and the original lithology.

Figure 16-1 Schematic geological section



Source: HZM, 2019

The weathering profile is characterised by extensive free silica generally in the form of veins, boxworks and massive zones of chalcedonic silica. The upper horizon has been modelled as an upper high-alumina and a lower low-magnesia domain. This is underlain by a high-magnesia horizon that has been modelled as an upper high-nickel (grades greater than 0.5% Ni) and a lower low-nickel (0.25% to 0.5% Ni) domain. Zones of ferruginous limonite (iron-rich zones with little silica) are present within the upper saprolite horizon.

Based on experience of site geologists, three geotechnical material type ranges were identified based on the silica content of the rock, as reported in the resource block model for geotechnical classification. The three material types are:

- Low silica material – silica content from 0% to 15%. This material type corresponds with what is generally known as ferruginous limonite.
- Medium silica material – silica content from 15% to 25%. This material type is generally known as siliceous limonite.
- High silica material – silica content greater than 25%. This material type is also generally known as siliceous limonite. The testwork performed by Golder (2004) and Snowden (2004d) indicates that this material is substantially more competent than the medium silica classification.

16.1.3 Density and moisture

Approximately 7,000 moisture content measurements have been taken at Vermelho; 4,065 from V2 and 2,810 from V1. The average moisture content at V2 is 13.40% and the average moisture content at V1 is 13.92%. Density values range from 1.08 t/m^3 to 2.22 t/m^3 at V1 and from 1.44 t/m^3 to 1.64 t/m^3 at V2, with average density values of 1.59 t/m^3 and 1.64 t/m^3 at V1 and V2, respectively.

16.2 Mine method

Typical truck and excavator mining using contractors is proposed. The proposed mining cycle would include the following primary activities:

- Grade control
- Clearing, stripping and stockpiling of topsoil
- Drill and blast (where necessary – see Section 16.2.3)
- Haulage, stockpiling and re-handling of ore and sheeting material
- Rehabilitation of mining areas with stockpiled topsoil.

16.2.1 Grade control

Due to the local variability of the orebody contacts, grade control drilling and assaying will be necessary to define the ore prior to presentation to the plant. Snowden completed a grade control study in 2004, which indicated that a 15 m x 15 m drill spacing is suitable for this deposit. For this study, HZM has elected to undertake grade control drilling on a 12.5 m x 12.5 m pattern which is consistent with other operations in the area. A RC drill rig with diameter between 4 inches and 5.5 inches is proposed. Samples will be collected at 1 m intervals. Drilling will be completed several benches ahead of mining. Routine grade control will be ensured by the presence of trained geotechnical staff, assisted as required by handheld XRF analysers.

16.2.2 Clearing and stripping

The entire mining area will be cleared any installations and vegetation to a depth of 0.3 m. All rubble, trees, bushes and other vegetable matter shall be stockpiled separately at designated locations. Vegetation suitable for use as fuel or commercial timber shall be stockpiled separately to all other cleared vegetation. The actual depth recovered will vary depending on location based on recommendations from the Environmental department. All areas where topsoil stripping occurs will be surveyed before and after topsoil removal. The topsoil will be pushed into piles by a dozer or grader before a loader or excavator is used to load it into trucks. Trucks will then haul the soil to stockpiles for later use on rehabilitation, or possibly direct to active rehabilitation areas. The stockpile areas will be cleared and surveyed prior to topsoil deposition.

16.2.3 Drill and blast

Golder (2004) reported that:

- 70% of the mined rock would be free dig
- 10% would be rippable
- 20% would require blasting.

This conclusion is based on a rock mass classification study and judgement based on Golder experience. Snowden completed additional work to estimate trafficability from a few tests carried out by Vale (Snowden, 2004b).

Based on the geotechnical material types above, the blasting requirements of the three material types are described as follows:

- The low silica material is free dig
- 25% of the medium silica material will require blasting
- All the high silica material will require blasting.

This classification is based on site experience and is more conservative than the numbers suggested by Golder. This methodology has the advantage of being more quantitative and being spatially orientated in the resource block model. Even with this more conservative approach, the drill and blast cost comprise less than 3% of the total mining cost and is therefore not considered a high risk to Project economics. The primary risk in this classification is in the medium silica range. The assumed 25% that will require blasting could be lower or higher. Considering the proportion of drill and blast cost in the total mining cost, it is reasonable to assume that the Project is not sensitive to this number.

Since majority of the material to be mined is oxidised rock, the average powder factor required to achieve acceptable fragmentation is estimated to be 0.20 kg/m^3 for the volume that is blasted. This will need to be confirmed in future studies.

16.2.4 Excavation guidance

In general, excavation guidance will involve the following steps:

- Ore control:
  - Dig plan creation
  - Mark out by surveyors
  - Face channel sampling of ore blocks
  - Ore spotting (in pit and excavator).
- Reconciliation:
  - Comparison of mined and processed with grade control and resource block models.

16.2.5 Excavation

If the guidelines in Section 16.2.3 are followed, excavator productivity should not be limited by the excavatability of the material. It might be useful to rip areas of localised high silica material or increase the powder factor by infill drilling. The conservative approach to blastability as described in Section 16.2.3 will also result in easier excavatability.

16.2.6 Trafficability

Based on the geotechnical material types, the trafficability of the three material types are described as follows:

- Low silica material should provide good trafficability in dry conditions, but is likely to deteriorate rapidly in wet conditions
- Medium silica material should hold up better in wet conditions
- High silica material should provide good trafficability in all conditions.

This classification is not quantitative but provides a comparative method of assessing trafficability through the mine life.

The risk of poor trafficability is that the active mining areas will become impassable and that mining may have trouble keeping up with the processing plant. This risk can be mitigated by:

- Sheeting access roads with appropriate, locally sourced competent material (e.g. beneficiation tailings or ferricrete).
- Implementing good drainage practices (e.g. drain working benches with a grader cut trench to the edge of the hill side and slope the active working face away from the excavator).
- Design working bench areas to assist water management. The proposed benches (100 m-wide strips) lend themselves to a standardised approach to drainage design on working benches.
- Maintain at least one month of ore on or near the ROM pad for contingency, particularly during wet conditions.

Snowden completed a review of testwork completed by Golder (Snowden, 2004b). This study concluded:

- siliceous limonite (high and medium Si) material will have satisfactory trafficability characteristics in moderate and/or intermittent wet conditions. ferruginous limonite (low Si) material is likely to have poor trafficability in such conditions. The trafficability of both materials is likely to deteriorate significantly in prolonged wet periods.
- The analysis has highlighted the need for well-graded material as running courses in haul roads, and most of the tested materials are reasonably well-graded.
- Significant improvements in the trafficability of the two materials may be achieved through the addition of competent coarse particles.
- Beneficiation rejects which include coarse siliceous material would be suitable for conditioning SapSil and ferruginous limonite for road construction.

An analysis of the silica distribution through the mine life indicates that the areas to be mined during the first six years of the mine life contain a majority of high silica material. This suggests that during the early life of the mine, trafficability will generally be better. This minimises the risk to Project economics and will allow for the development of appropriate mitigation strategies.

16.2.7 Tipping

Ore will be hauled from the pit to stockpiles or to ROM. Ore from the long-term stockpiles is then re-handled and hauled from these stockpiles to the ROM for further re-handle into the plant.

Waste will be placed on external waste dumps. Pits are not intended to be backfilled at this stage, as the waste component for this Project is low and mineralised material is at the base of the pits which may be economic. Backfilling pits remain an opportunity for the Project to reduce mining costs.

16.2.8 Mine to mill strategy

The primary aim of this mining operation will be to maximise margin, particularly in the early life of the mine, and to supply ore to the processing plant so it that maintains a constant acid consumption below 350 kg/t HPAL feed. The mining layout and methodology relies on a combination of mining selectivity and stockpiling. The stockpiling strategy proposed in this study relies on dividing ore into different stockpiling ranges to maximise nickel grade and control acid consumption. The stockpile categories are based on material type and profit per tonne (PPT), or revenue minus processing and sales cost. The material type (primarily limonite and saprolite) controls the magnesium/acid levels. The highest PPT categories will primarily be short-term stockpiles which can be managed on or near the ROM stockpile area. The lower PPT and higher acid consumption material will be placed on larger stockpiles to be fed to the processing plant at the end of the mine life. The processing plant will also use a seven-day equivalent blended stockpile of crushed ore, which will assist in reducing variations in the plant feed.

16.2.9 Rehabilitation

Surface waste dump landforms at each of the various mining locations will be progressively rehabilitated during their construction. Once a mining location is completed, the upper surface of the waste dump will be rehabilitated. Topsoil will be sourced directly from the mining areas or from topsoil stockpiles.

The following activities were allowed for the rehabilitation of waste dumps:

- Push down waste dump batters to final formation
- Form waste dump top to final formation
- Load topsoil and dump at rehabilitation area within 1,000 m of source
- Spread topsoil onto battered slopes to minimum thickness of 300 mm
- Rip and seed waste dump battered slopes
- Spread topsoil onto waste dump top to minimum thickness of 300 mm
- Rip and seed waste dump topsoil.

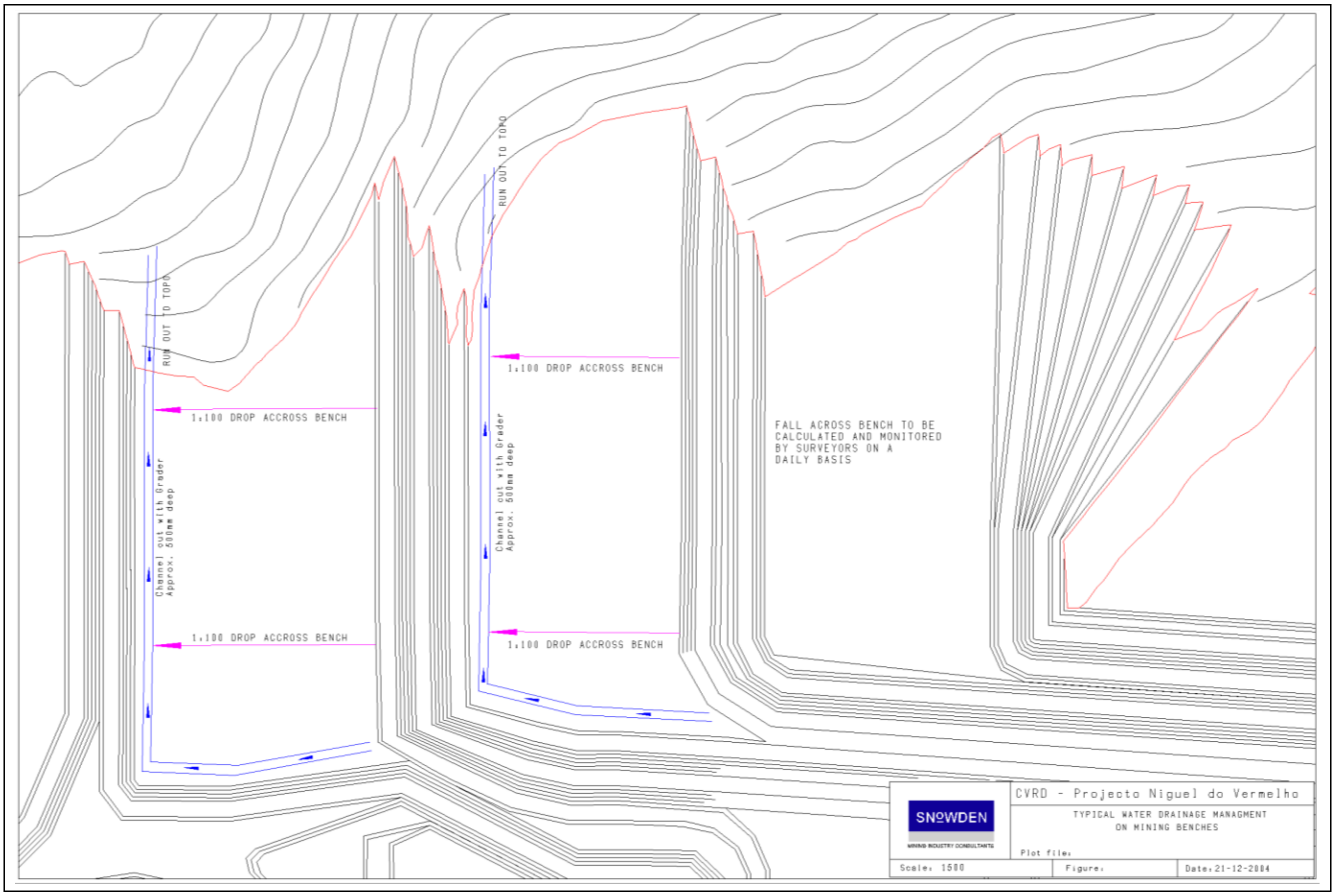
16.2.10 Water management

The shape of the deposits (two hills) in this high rainfall region will assist with water management. It should generally be possible to drain water off the hillsides.

The proposed layout is a series of 100 m-wide strips inside each stage. The advantages of this layout include:

- Each mining bench (or strip) can be accessed from the edge of the hill or pit.
- Drainage is simple, the water can be led along a grader cut trench to the edge of the hill and away from the working area as shown in Figure 16-2.
- The individual strips make up small enough units for scheduling purposes. The material on a 4 m bench within the area of a mining strip is on average close to a week’s production.
- By benching the strips down the sides of the hill, a very orderly operation can be maintained while ore can be accessed at various points along the length of the exposed strip. This allows for very good selectivity.

Figure 16-2 Typical drainage layout



Source: Snowden, 2004b

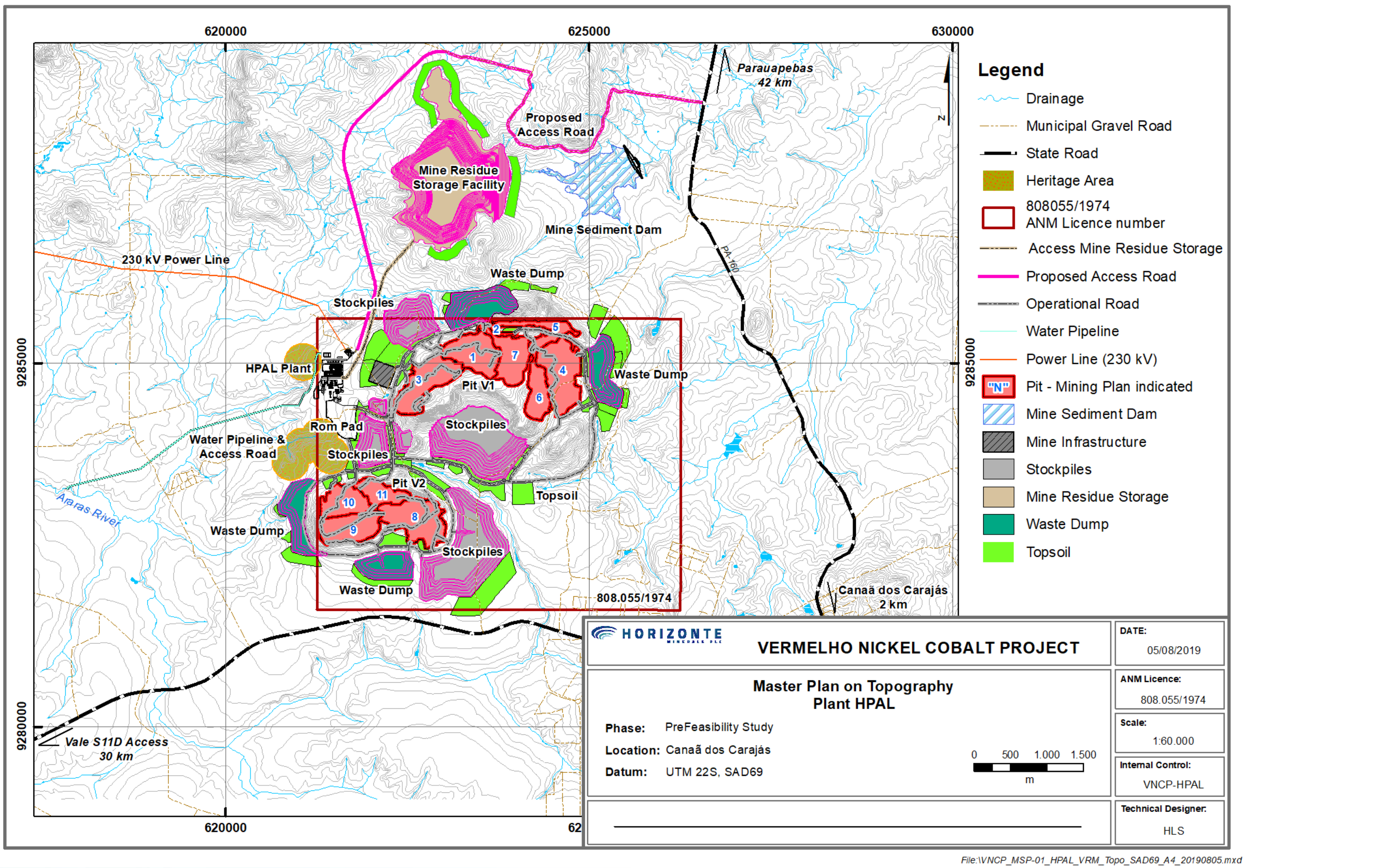
16.3 Site layout

16.3.1 Overall site layout

Roads, dumps and stockpiles were preferentially located to minimise haulage distance. In addition, dumps were generally located outside the RF 2 unconstrained Inferred optimisation pit shell, to ensure potential future resources are not sterilised.

The general site layout is shown in Figure 16-3.

Figure 16-3 Planned site layout



Source: HZM, 2019

16.3.2 Clearing and topsoil stripping

Clearing and topsoil stripping were estimated based on an average strip depth of 300 mm. Large areas are required for topsoil stockpiling as topsoil stockpile heights were limited to 1.5 m. Table 16-1 summarises the clearing areas (in hectares) and topsoil volumes (in kilo bank cubic metres or kbcm).

Table 16-1 Clearing areas and topsoil volumes

Deposit/Type	Area (ha)	Topsoil produced (kbcm)
Pit	302	906
Waste dumps	135	406
Roads	44	133
Infrastructure	9	27
Ore stockpiles	278	834
Topsoil stockpiles	185	0
Total	**954**	**2,306**

16.3.3 Dumps

Table 16-2 summarises the dump design parameters. Dump heights were not restricted.

Table 16-2 Dump design parameters

Parameter	Unit	As-tipped
Batter angle	°	37
Lift height	m	10
Berm interval	Vertical m	10
Berm width	m	27.5
Overall slope – toe to toe, no ramp	°	13.8

Table 16-3 summarises the dump capacities. Waste was swelled by 25%.

Table 16-3 Dump capacities by deposit

Deposit	Dump type	Capacity (klcm)
V1	Dump 1, 2, 3 and 7	6,473
	Dump 4 to 6	4,447
V2	Dump 8 and 11	4,501
	Dump 9 and 10	3,698

16.3.4 Ore stockpiles

Long-term stockpiles will be large and were designed using the same parameters as the dumps (Table 16-2).

16.3.5 Haul roads

Approximately 32 km of haul roads are planned. This will required cut and fill volumes of about 450 kbcm. Some roads are required to cross small water courses and will require culverts to allow water flow under the road. This should be assessed during subsequent studies.

16.3.6 Rehabilitation

Areas will be rehabilitated progressively as mining permits.

16.4 Life of mine schedule

16.4.1 Methodology

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 (NPV) in the presence of physical quantity and grade constraints.

16.4.2 Parameters and constraints

Mining inventory

The mining inventory is reported within the designed pits. Table 16-4 summarises the mining inventory by deposit.

Table 16-4 Mining inventory by deposit

Deposit	V1	V2	Total
Total ore (Mbcm)	50.8	40.5	91.3
Total ore (Mt)	77.6	63.7	141.3
Ni (%)	0.92	0.90	0.91
Co (%)	0.05	0.05	0.05
Fe (%)	26.2	19.4	23.1
Al (%)	0.95	0.60	0.79
Si (%)	19.5	24.1	21.5
Mg (%)	3.45	4.25	3.81
Total concentrate (Mt)	42.4	30.1	72.5
Mass pull (%)	54.6%	47.3%	51.3%
Ni (%)	1.31	1.50	1.39
Co (%)	0.07	0.07	0.07
Fe (%)	37.5	32.2	35.3
Al (%)	1.19	0.89	1.06
Si (%)	9.7	11.8	10.6
Mg (%)	4.23	5.99	4.96
Total waste (Mbcm)	**6.8**	**5.2**	**12.1**
Total waste (Mt)	**11.1**	**9.2**	**20.2**
Total (Mbcm)	**57.6**	**45.7**	**103.4**
Total (Mt)	**88.7**	**72.9**	**161.6**
Strip ratio (waste:ore)	**0.14**	**0.14**	**0.14**

Material types

The mining model was coded with material types (geological and geotechnical). Within these material types, additional bins for acid consumption (in 100 kg/t increments) and block profitability (in US$50/t increments) were incorporated to allow ore and waste scheduling selectivity (e.g. stockpiling lower grades and blending acid).

Resolution

The schedule was completed in annual increments over the life of the Project.

The mining inventory was separated into pit stages and 8 m vertical benches for scheduling. Within each bench, material was separated into the material types for grade maximisation or blending purposes.

Active mining areas

The schedule minimised the number of pit stages mined in any period to simplify the mining operation.

The timing of each mining area was determined by first running an unconstrained schedule (with no limit on active mining areas) and analysing the results. The mining areas and deposits were then restricted to specific timeframes.

Bench turnover

A vertical rate of advance of approximately 48 m per year was applied for scheduling.

Mining

A maximum mining rate of 12 Mt/a was determined through scenario analysis.

Processing throughput and ramp-up

The initial process throughput is 1 Mt/a HPAL feed and 350 kt/a acid plant, both doubling in capacity after three years of processing. First metal production is planned for 2023. A ramp-up of 75% of full production was allowed for in the first year of each step of production. The beneficiation circuit throughput was allowed to vary, to meet the HPAL feed capacity.

Grade constraints

For scheduling, the following grade constraints were applied:

- Magnesium content ≤5%
- Acid consumption ≤350 kg/t HPAL feed.

Economic assumptions

The economic assumptions applied for scheduling (driving the discounted value calculation) were the same as for pit optimisation.

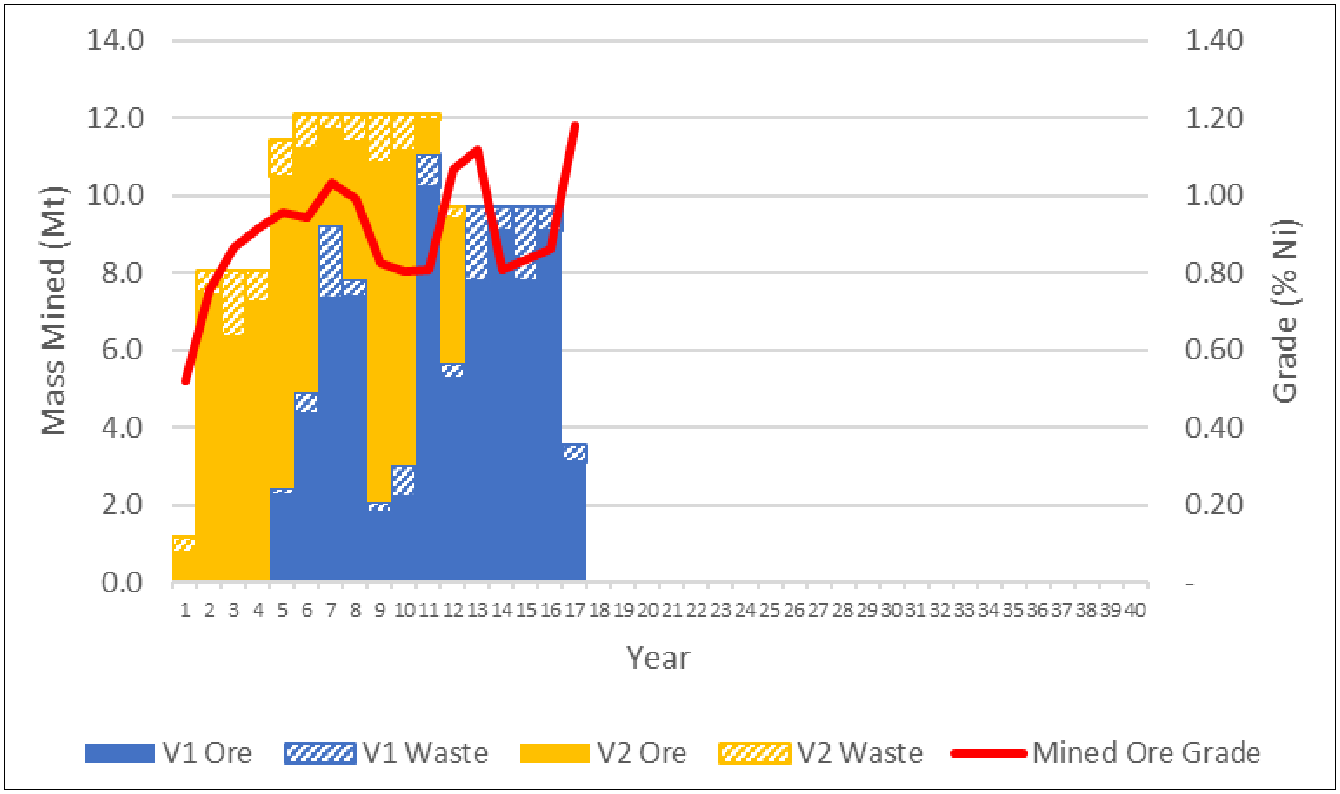
16.4.3 Schedule results

While the schedule is at either monthly or quarter scale, the results are presented annually for period consistency and clarity. Year 1 corresponds to calendar year 2023.

Mining schedule results

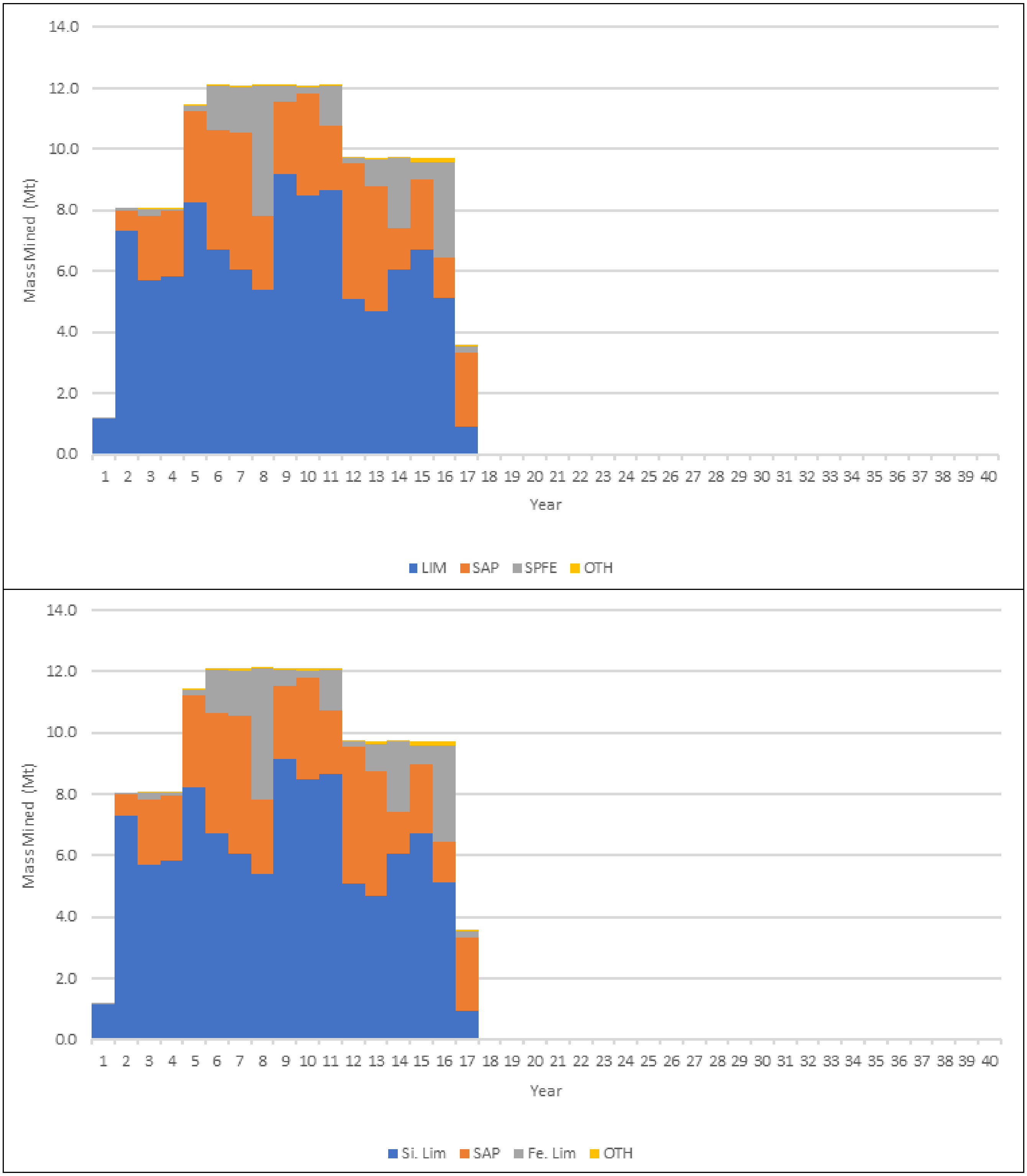
Figure 16-4 shows the total movement by deposit and Figure 16-5 shows total movement by rock type. Mining commences at V2 at a rate of approximately 8 Mt/a. V1 is introduced in year 5 and the mining rate increases to approximately 12 Mt/a with mining from both deposits. V2 is completed in year 12 and mining continues at V1 until year 17 at a lower mining rate of just under 10 Mt/a. Mining is completed in 17 years, accessing all the higher-grade ore and building a larger lower-grade stockpile for depletion over the remaining LOM.

Figure 16-4 Total ex-pit movement by deposit



Source: Snowden, 2019

Figure 16-5 Total movement by rock type (top – lithology, bottom – geotechnical rock types)



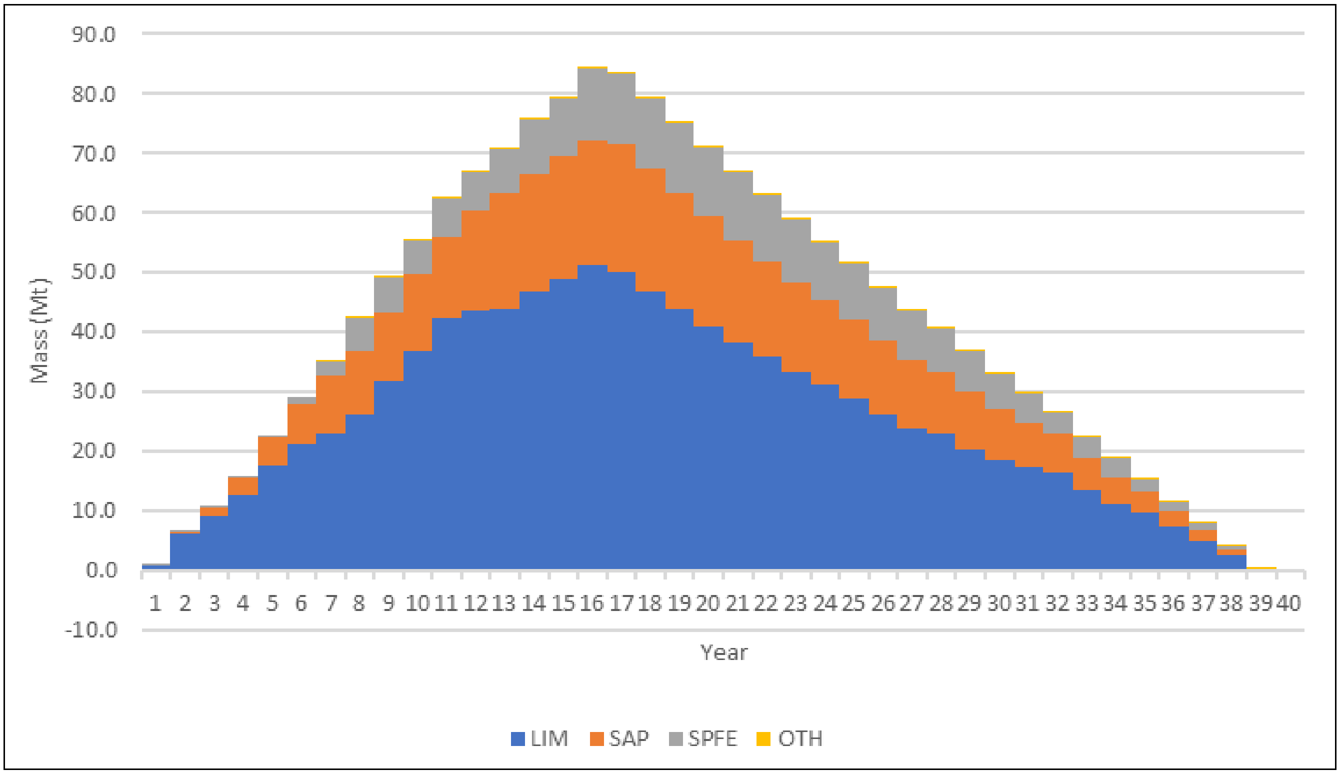
Source: Snowden, 2019

The active mining areas were maintained to three at any time to minimise complexity and excavator movement. In some years, the number of active deposits increases to four, due to movement between deposits during the year.

Long-term stockpile scheduling results

A large strategic stockpile is built; allowing high-grade material to be preferentially processed and, in the short term, allows deleterious material to be blended. The peak stockpile size of 84 Mt occurs in year 16, and then is depleted over the remaining mine life (Figure 16-6).

Figure 16-6 Long-term stockpile balance and movements by material type

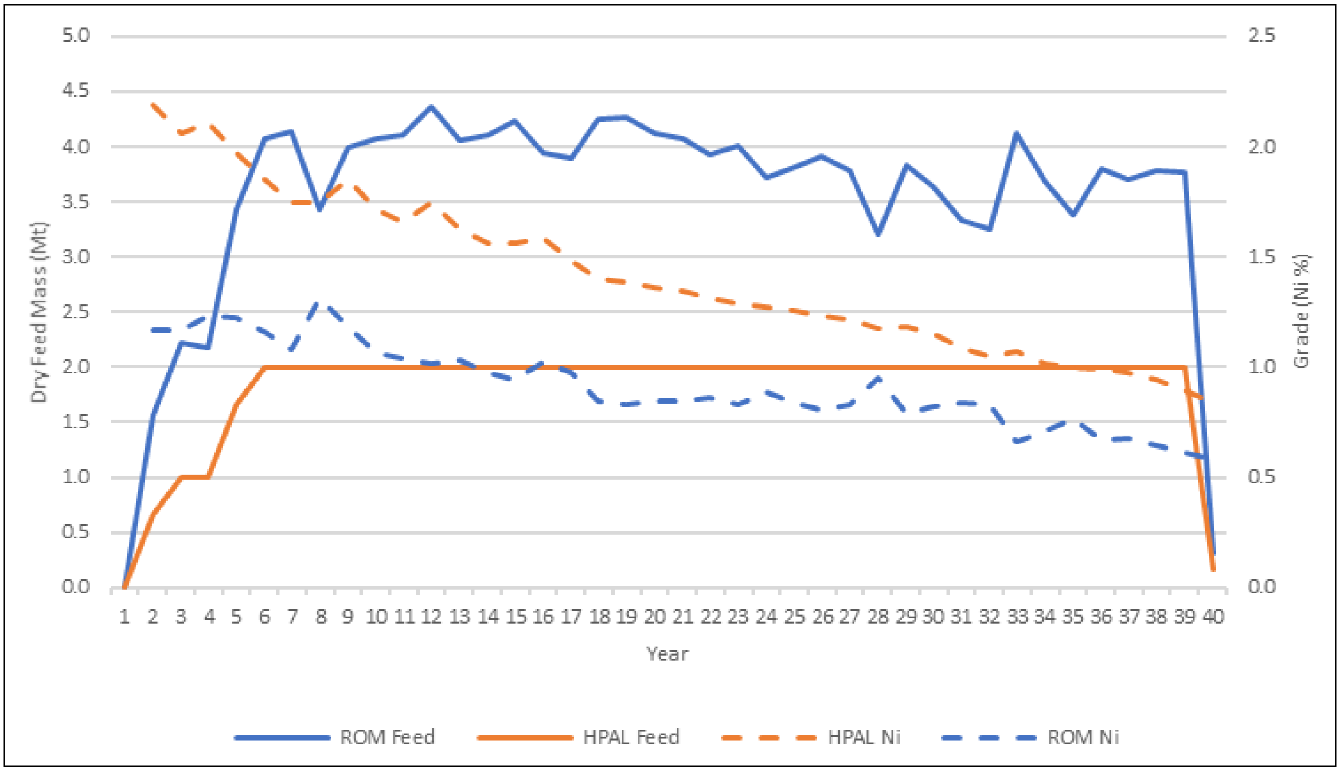


Source: Snowden, 2019

Processing schedule results

Figure 16-7 shows a high-level summary of the processing for the Project. The HPAL plant capacity of 2 Mt/a of feed is met throughout the mine life after ramp-up. Additionally, the acid capacity is constant (as a ratio of HPAL feed) over the life as the feed is blended for acid consumption. The nickel grade (after beneficiation) shows a declining trend over the mine life, consistent with NPV maximisation, starting at approximately 2.0% Ni before reducing to approximately 1.0% Ni at the end of the mine life. The beneficiation circuit utilisation varies over the life according to the silica content of the feed and the resultant mass yield, and peaks at approximately 4.4 Mt/a.

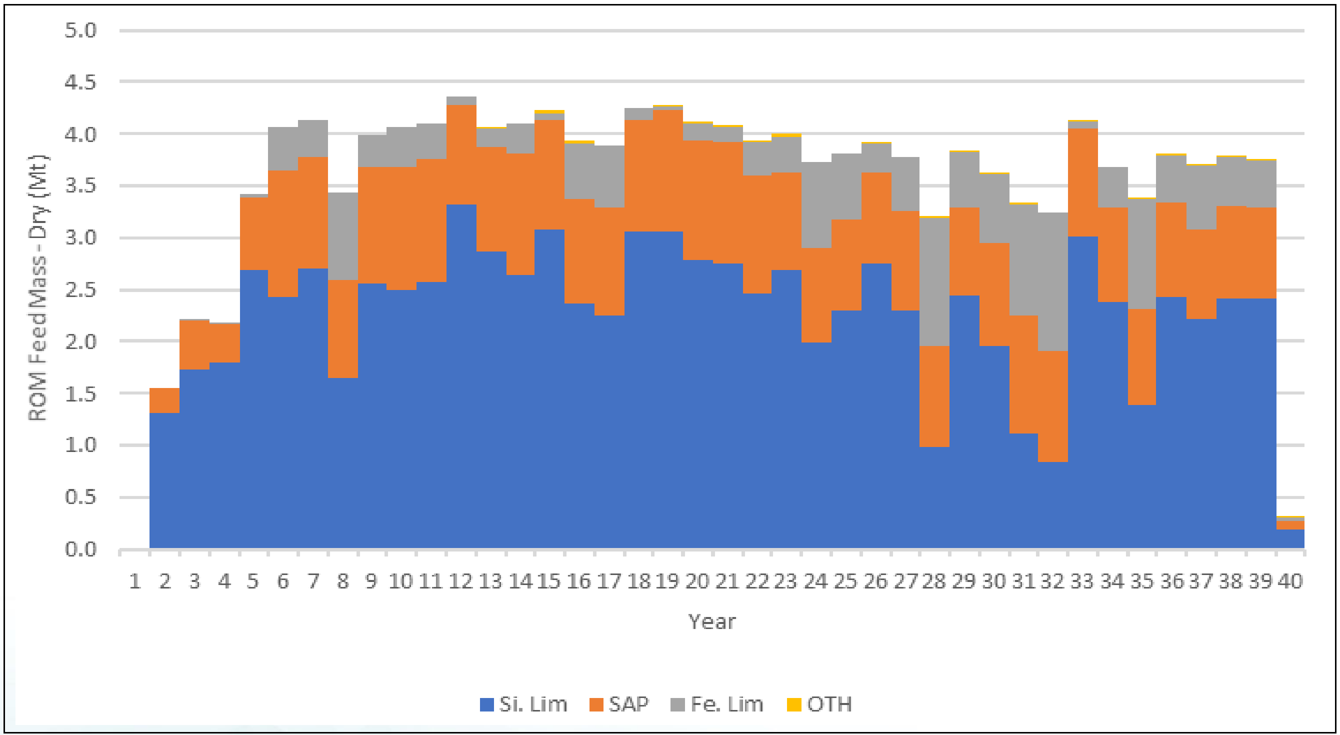
Figure 16-7 Processing schedule summary



Source: Snowden, 2019

Figure 16-8 shows the ore feed to the beneficiation circuit by rock type. Although not a requirement, the split between limonite and saprolite is consistent on an annual scale as the magnesium content is blended.

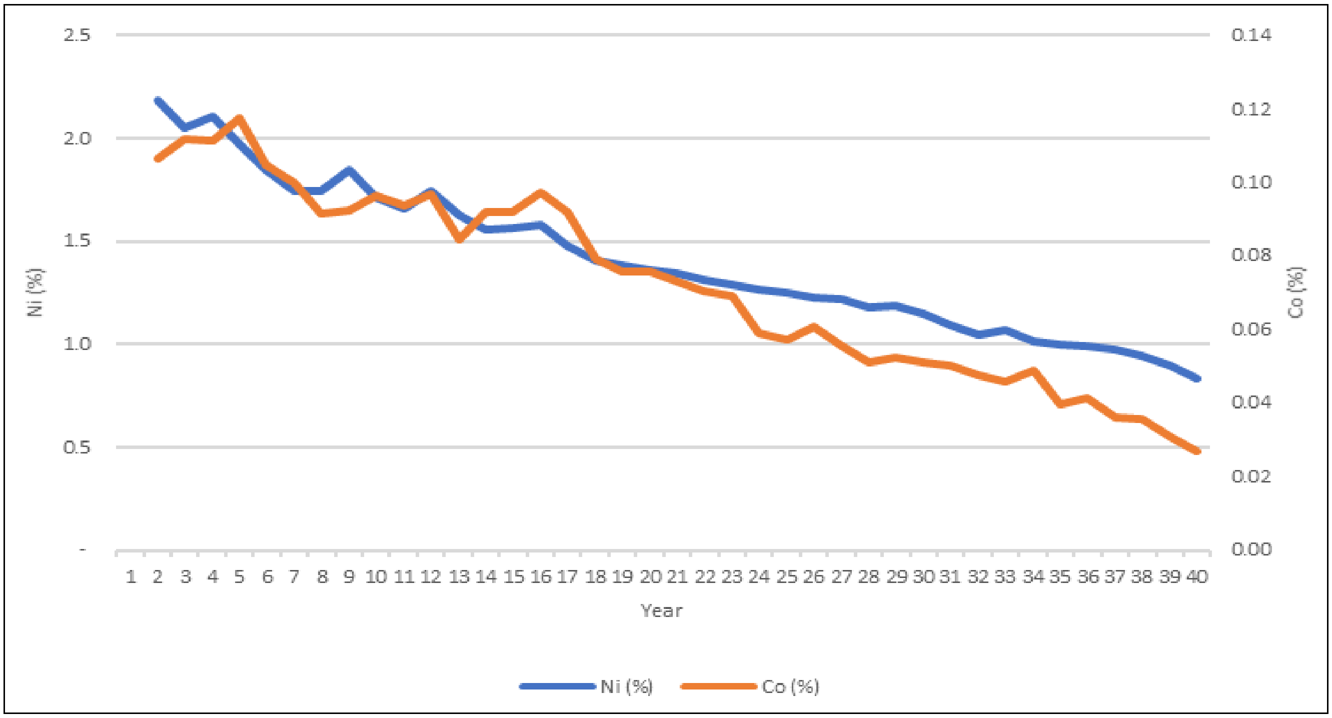
Figure 16-8 Ore feed by rock type



Source: Snowden, 2019

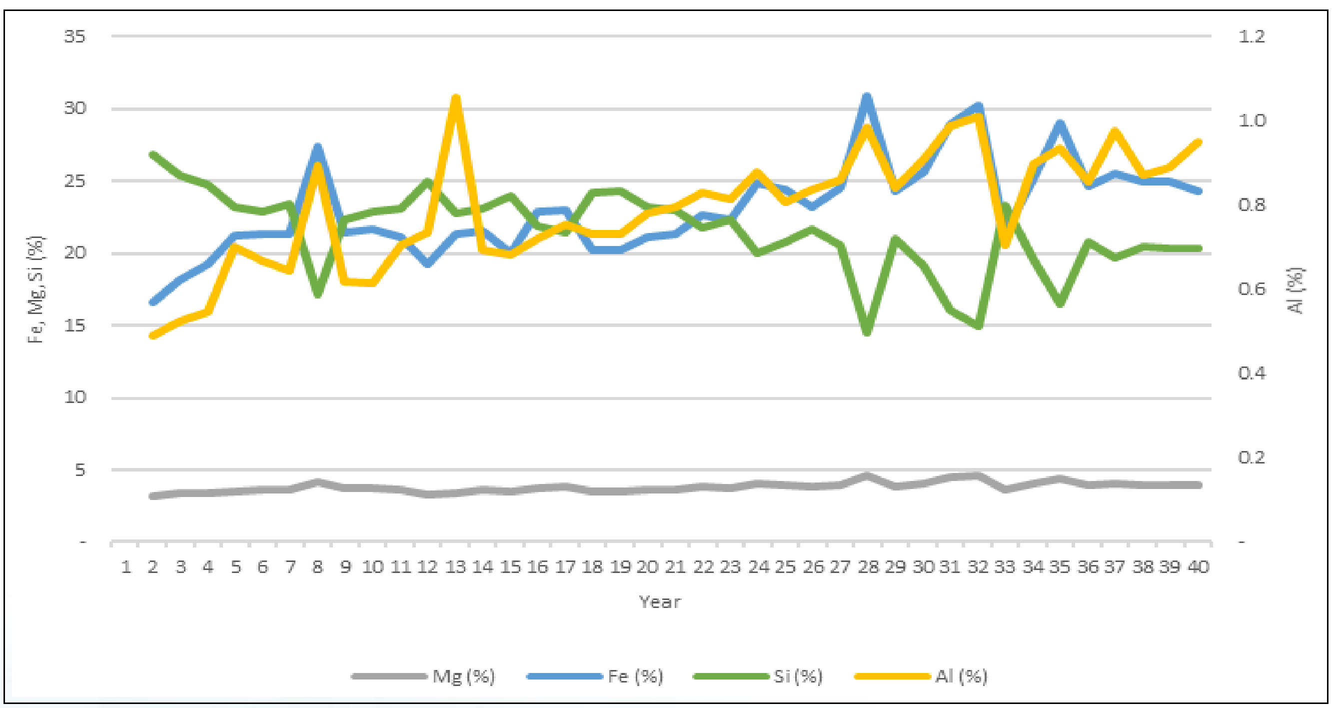
Figure 16-9 illustrates that the HPAL nickel and cobalt grades decline over the life of the Project. The magnesium content in HPAL remains steady at approximately 5% over the life of the mine (Figure 16-10), maintaining a fixed acid consumption rate of 350 kg/t HPAL feed and mitigating the need for additional lime.

Figure 16-9 HPAL feed grades (Ni and Co)



Source: Snowden, 2019

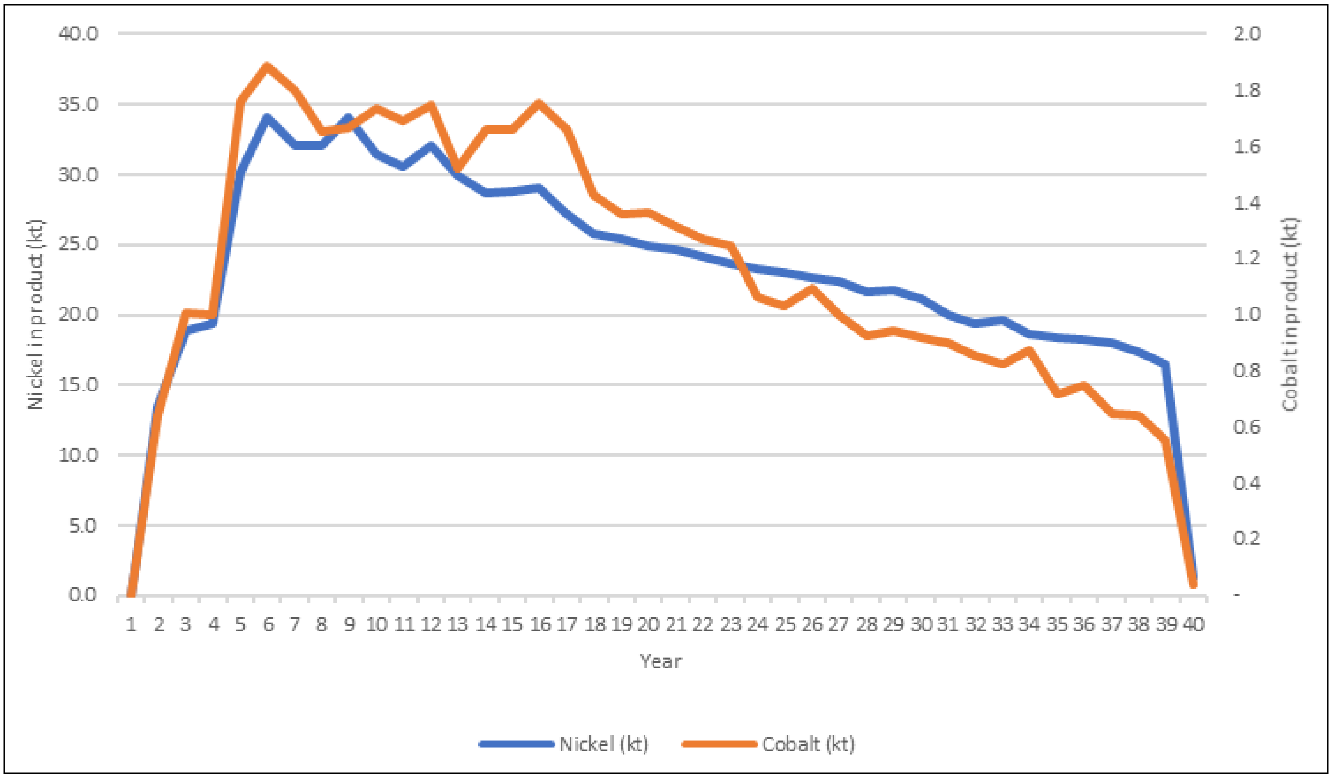
Figure 16-10 HPAL feed grades – other elements



Source: Snowden, 2019

The nickel production peaks at 34 kt/a in year 9 (Figure 16-11). The cobalt production peaks at 1.9 kt/a in year 6.

Figure 16-11 Product schedule



Source: Snowden, 2019

16.5 Mining requirements

16.5.1 Battery limits

Battery limits for the mining study were limited to those activities required to define, excavate and transport ore to the process plant:

- Grade control drilling and assaying of the orebody ahead of excavation
- Constructing mine haul roads
- Clearing of vegetation from forests and paddocks within the footprint of the mining workings
- Removing and stockpiling topsoil and ferricrete for sheeting
- Excavating waste material and stockpiling in permanent surface waste dumps
- Excavating ore and trucking to a surface stockpile adjacent to the excavation
- Re-handling ore from the stockpile to the ROM
- Re-shaping waste dumps to create a stable landform
- Re-handling topsoil from topsoil and spreading on reshaped waste dumps
- Managing and coordinating the excavation activities to provide the process plant with ore feed within specified limits.

In addition to the above, provision was included for the transport of beneficiation rejects and residue from the process plant to the residue storage facility.

There were many operational interfaces that were excluded from the mining studies, including:

- Ex-pit dewatering
- HZM mining team buildings and infrastructure, and general administration of the site.

16.5.2 Operating philosophy

To mitigate production risk and initial capital burden, the mining operations will be outsourced to an experienced mining contractor. It was envisaged the contractor would operate under a conventional Schedule of Rates style contract which may include both fixed and variable components. The contract would be structured around achieving the mining schedule or revisions advised by HZM from time to time throughout the Project execution, as well as a variety of non-commercial objectives associated with safety, environment and quality. The details of this arrangement were not considered in this study.

The costs and requirements were based on contractor budget submissions for the Araguaia nickel project as part of the 2018 FS (HZM, 2018).

HZM intend to employ a small mining team comprising of senior management and technical services. The mining team will oversee and coordinate the execution of the work and manage the performance contractor on a daily basis.

16.5.3 Operating hours

The shift schedule will be a 24-hour day, seven-day week roster. The daily roster will consist of a nine-hour dayshift, a nine-hour afternoon shift and a six-hour nightshift (to be consistent with the nearby Carajás operation). The average work week will be 42 hours with four crews. During each year, there are several public holidays that will be observed, and it is expected that a significant amount of time will be lost to wet weather. An estimate of this lost time due to weather was based on data supplied by Golder and collected at the Sossego mine site (approximately 20 km from Vermelho) and presented in Table 16-5.

Table 16-5 Local rainfall analysis

Rainfall per day (mm)	No. of events in sample	Average no. of events per year	Estimated workdays lost per event	Estimated average no. of workdays lost
>100	1	0.2	2	0.4
>50<100	9	1.7	1	1.7
>20<50	77	14.8	1	14.8
>10<20	101	19.4	0.08	1.6
<10	504	96.9	0.00	0.0
Total number of workdays lost (average)				**18.5**

16.5.4 Equipment

A fleet of 22 m^3 8x4 on-road trucks was used with 48-t excavators. On-road trucks (22 m^3 8x4) and a 23-t front-end loader will be used for rejects handling. The 20-t drills were selected. The peak equipment requirements are presented in Table 16-6.

Table 16-6 Peak equipment requirements

Item	Peak quantity
Dump truck, Scania G440, 8x4HT, 22m³, with Retarder Breaking	123
Loader, Caterpillar 966L	15
Excavator, Caterpillar 349D L	13
Dozer, Caterpillar D-6N	7
Motor grader, Caterpillar 140-K	2
Water truck, MB-2423, 20.000l	3
Tandem vibratory compacting roller, Dynapac CA-25/CA-25A	1
Tractor, Valtra/Valmet A950 4x4	1
Truck, MB-1620, rigid, with 6-t Crane	1
Drill rig	5

16.5.5 Manning

Mining contractor manning was calculated for each year of production. Table 16-7 shows the peak by role. The numbers included for operators will vary with the numbers of equipment and as a result, the total number of employees will vary through the life of the mine.

Table 16-7 Peak mining manpower by role

Role name	No.	Role name	No.
Operator	552	Office assistant	4
Loader operator	66	Secretary	1
Excavator operator	56	Worker (general help)	6
Helper	16	Warehouse supervisor	1
Dozer operator	7	Warehouse assistant	4
Motor grader operator	2	Buyer	1
Compactor roller operator	1	Driver (admin)	3
Tractor operator	1	Quality control technician	1
Backhoe operator	1	Contract manager	1
Drill operator	22	Production supervisor	5
Chief engineer	0	Surveyor	3
Resident engineer	1	Grader	3
HSM engineer	1	Surveyor assistant	12
Mining engineer	1	Maintenance supervisor	1
Site medical doctor	1	Mechanic (general)	10
Control room supervisor	1	Maintenance assistant	10
Measurements technician	4	Mechanic (lubrication)	10
Cost analyst	3	Mechanic (tyre handling)	10
Truck dumping helper	8	Driver	9
Technicians assistant	2	Mechanic (electrical)	8
Cost assistant	2	Daytime security officer	10
General help	1	Night-time security office	10
Nurse	1	General help – maintenance	5
Nursing technician	2	General help – lubrication	5
HSM technician	5	Electrician	5
Administrative supervisor	1	Bus driver	5
Personnel supervisor	1	Welder	4

16.5.6 Consumables

The main consumables for mining are fuel and explosives. Peak diesel requirement is estimated to be 1,646 kℓ/a and peak explosive requirement is estimated to be 870 t/a.

17 RECOVERY METHODS

17.1 Introduction

Based on the metallurgical testwork, the mine schedule and other supporting data provided, Simulus developed engineering drawings along with capital and operating cost estimates for the processing facility in addition to a Project Implementation Plan. An overall estimate of the plant cost and time to construct and commission the plant was completed.

Snowden notes that all tabulated parameters relating to Section 17.3 (Process description), including crushing and HPAL operations, are based on a 7,446 hours per annum operation (i.e. 85% overall plant uptime).

17.2 Process selection

The process selected for the Project is the production of separate high-purity nickel and cobalt sulphate products via high-pressure acid leaching (HPAL), MSP, POX of the MSP, SX of the leached metals in the POX discharge and crystallisation of the final nickel and cobalt sulphate products and the kieserite by-product.

The previous project owners had evaluated multiple flowsheet options during the previous PFS, including:

1) HPAL and MHP production.
2) HPAL and MHP, ammonia re-leach (AR) and carbonate product.
3) HPAL and MHP, AR and nickel cathode and cobalt sulphide product.
4) HPAL and MHP, AR and nickel cathode and cobalt cathode product.
5) HPAL and MHP, AR, nickel carbonate and cobalt sulphide product.
6) HPAL and MHP, AR, nickel oxide and cobalt sulphide product.
7) HPAL and MHP, AR, nickel cathode from oxide and cobalt sulphide product.
8) HPAL and MSP production.
9) HPAL, MSP, POX and hydrogen reduction to make nickel and cobalt briquettes.

The previous owner had selected Option 4 as the preferred flowsheet option from the above list. Option 9 had the second-best Project NPV and IRR, due to the marginally higher capital costs compared to Option 4. However, since 2002 (when the previous PFS was completed), a new market for battery-grade nickel and cobalt sulphate has emerged. These battery precursor products attract a premium to nickel and cobalt metal prices.

The current Project is a variation of Option 9 and replaces the hydrogen reduction and briquetting steps with crystallisation. The reduced capital cost of crystallisation versus hydrogen reduction and briquetting and the price premium for battery-grade sulphate products make this option more attractive. To this has been added a kieserite crystallisation step to remove Mg from the process water and make a potentially saleable by-product.

It also has the benefit that the intermediate mixed sulphide may be more marketable and economically attractive as a product comparted to the mixed hydroxide. The nickel/cobalt grade in the MSP concentrate is higher, which reduces shipping costs, and the number of potential customers is greater. Mixed sulphides also contain fewer impurities than mixed hydroxides and are thus simpler to convert through to nickel and cobalt sulphates.

17.2.1 Process overview

The Project has been designed to process 4.34 Mt/a of ROM ore. Of this material, 2.34 Mt/a is rejected as coarse, low grade siliceous waste from the beneficiation plant and 2 Mt/a is fed to the HPAL processing plant as an upgraded concentrate (1 Mt/a per stage). An overall block flow diagram is provided in Figure 17-1 and the proposed plant layout in Figure 17-2.

The processing plant consists of the following main process unit operations:

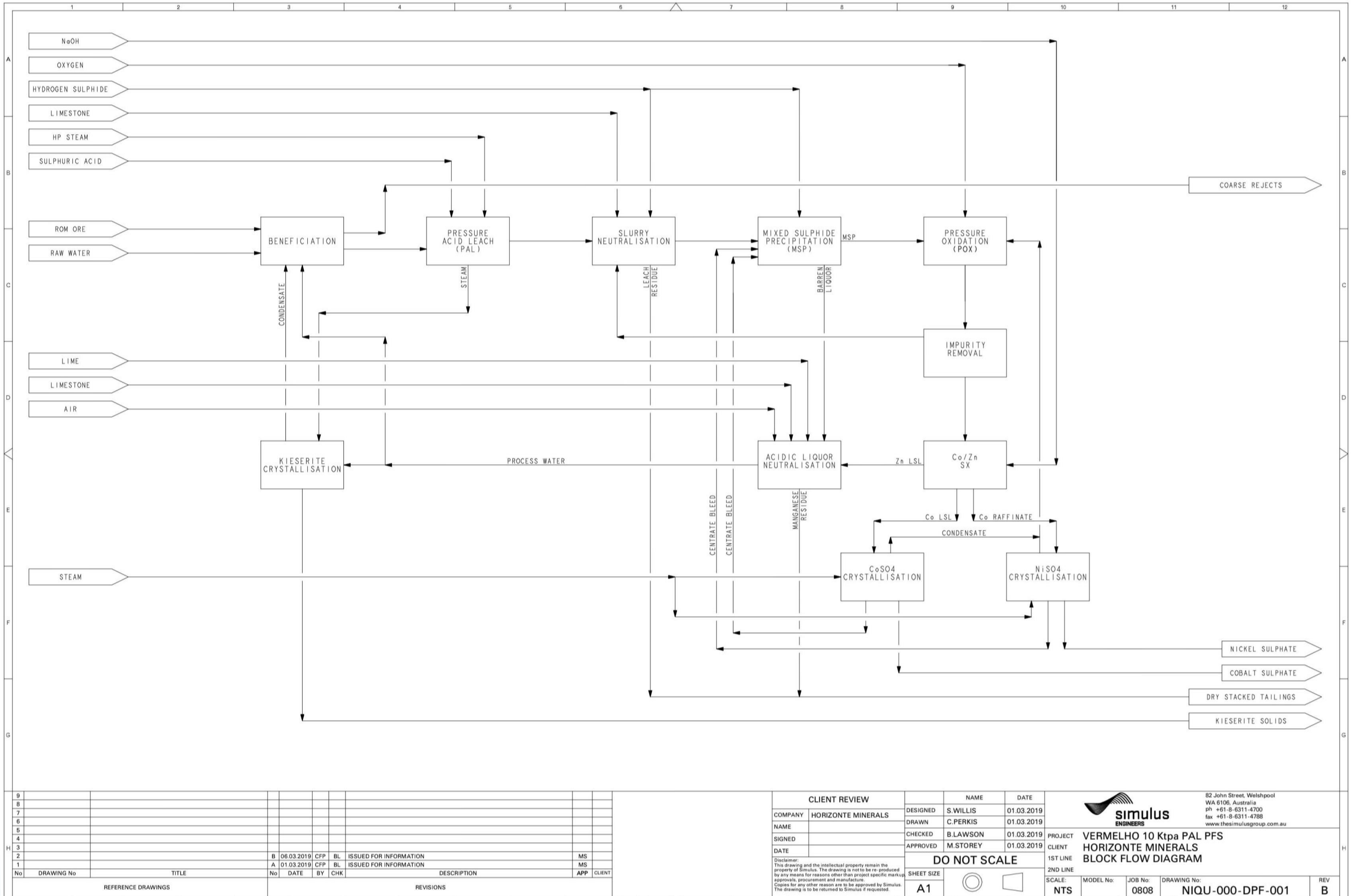
- Beneficiation
- HPAL
- Slurry neutralisation and residue filtration
- MSP
- POX
- Impurity removal
- CoSX
- Nickel sulphate crystallisation
- Cobalt sulphate crystallisation
- Acid liquor neutralisation
- Kieserite crystallisation
- Sulphuric acid plant
- Reagents and utilities.

The Project Implementation Plan includes a staged approach which minimises initial financing requirements; whereby cash flow from the first stage is used to fund the second stage expansion. Therefore, the processing plant has been designed in two stages. The first stage begins operation in year 1 and the second stage comes online in year 4. Each stage is designed to process 1 Mt/a of concentrate through the HPAL autoclave and downstream plant up to the production of the MSP intermediate product.

A single refinery treats the MSP from both stages through to final nickel and cobalt sulphate products. The justification for the single refinery train is the smaller physical size and therefore minor additional capex required for the full production, but also that the nickel grade is higher in the early years of operation and more than half of the refining capacity would have been required making two identical trains impractical.

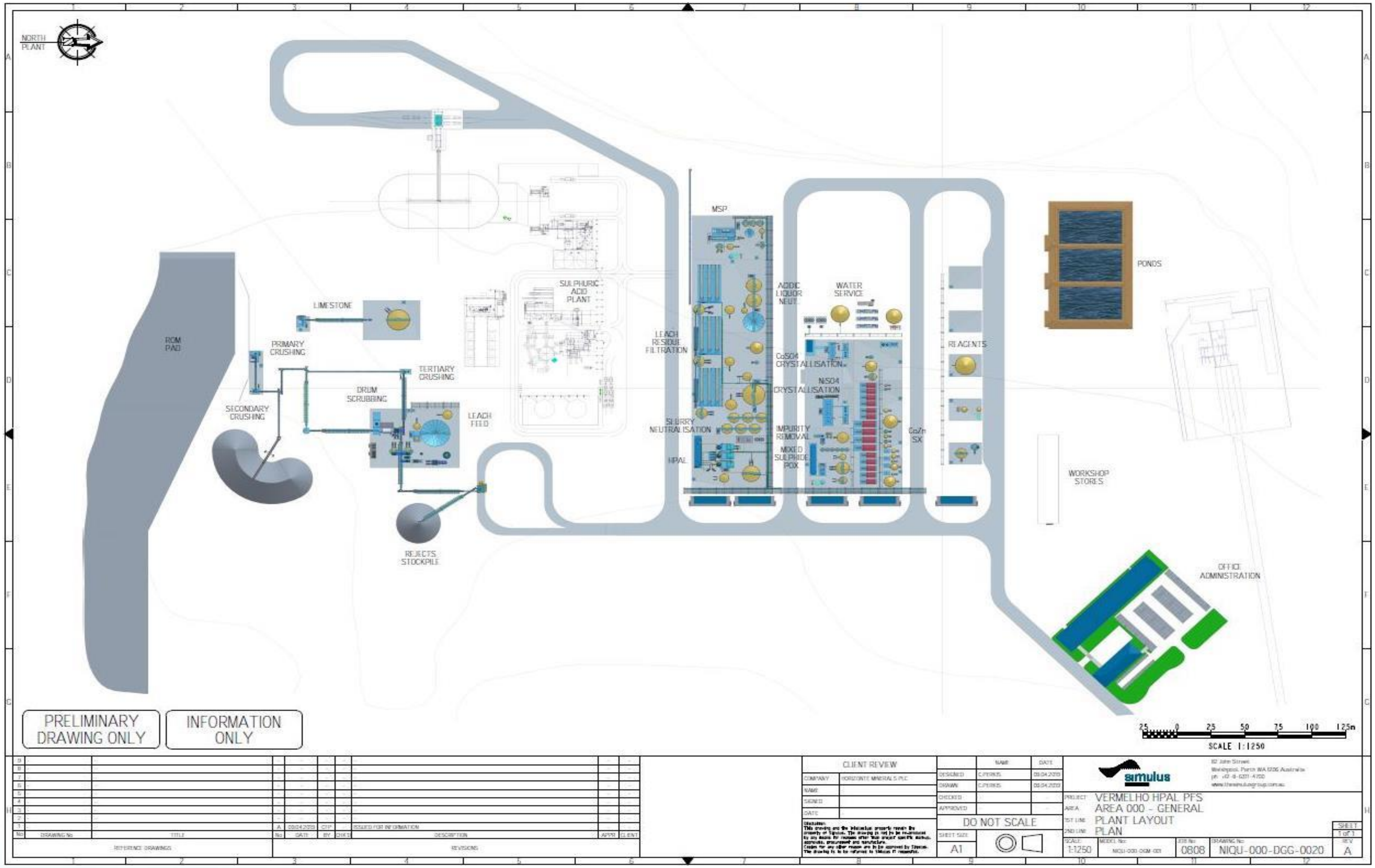
The crushing area has been designed for 75% availability. The remaining processing plant has been designed with 85% availability. All slurry pumps have duty and standby pumps installed, but liquor pumps only have a duty pump installed. Details of each plant area are given in the following sections.

Figure 17-1 Process block flow diagram



Source: Simulus, 2019

Figure 17-2 Plant layout



Source: Simulus, 2019

17.3 Process description

17.3.1 Beneficiation

The beneficiation process upgrades ROM nickel ore by rejection of free silica to produce a HPAL feed concentrate. There are two trains of beneficiation, each of which includes:

- Crushing in toothed roll crushers
- Scrubbing and washing to liberate nickel-bearing fines
- Separation of the coarse siliceous waste by screening
- Attritioning of undersize material
- Separation of the attritioned fines with hydro-cyclones and high-capacity, high-frequency screens.

The outcome of the overall beneficiation process is to separate the fine limonite and nickel-bearing clays from the coarser barren silica waste with a grain size greater than 0.15 mm. The coarse silica waste goes to a waste rejects stockpile and the concentrate is fed to the processing plant.

ROM trucks unload ore from either operating pits or temporary storage piles onto a fixed 1 m grizzly. A hydraulic rock breaker is provided to reduce any grizzly oversize. Grizzly undersize is fed to the primary toothed roll crusher by variable speed apron feeder. The crusher discharges onto a conveyor fitted with weightometer and tramp magnet which feeds the secondary toothed roll crusher. Secondary crushed ore is sampled and conveyed to a telescopic, radial stacker and stockpiled. The stockpile provides blending and surge capacity between crushing and the downstream beneficiation and processing plants. A diverter chute and stockpile bypass conveyor are also included to feed the beneficiation area directly.

Secondary crushed ore is reclaimed by front-end loader and conveyed with tertiary crushed ore to the beneficiation surge bin. An apron feeder is used to control the ore feed rate as measured on the weightometer at the drum scrubber feed conveyer.

Key criteria of the crushing section are shown in Table 17-1.

Table 17-1 Crushing operating parameters

Item	Unit	Value
ROM ore feed rate per train	dt/hr	291
ROM ore moisture content	%w/w	20.6
Crushing area availability	%	75
Primary crusher type	text	Toothed roll crusher
Primary crusher top size	mm	1,000
Primary crusher P80	mm	300
Primary crusher work index	kWh/t	12
Secondary crusher type	text	Toothed roll crusher
Secondary crusher top size	mm	400
Secondary crusher P80	mm	120
Secondary crusher work index	kWh/t	12
Tertiary crusher type	text	Toothed roll crusher
Tertiary crusher top size	mm	150
Tertiary crusher P80	mm	40
Tertiary crusher work index	kWh/t	12
Crushed ore stockpile capacity	days	7
Consumed power	kW	670

The drum scrubber discharges onto a double deck, vibrating, banana screen, the oversize from which goes to the tertiary roll crusher. The undersize from the banana screen is directed to a grit screen. The grit screen oversize, banana screen mid-size fraction and dewatering screen oversize are combined on the rejects conveyor. The combined rejects are then washed with kieserite crystalliser condensate or raw water on the rejects screen to remove soluble magnesium sulphate before being directed to the rejects bin or temporary stockpile. The rejects are trucked to the rejects waste stockpile.

The grit screen undersize is then fed to primary hydro-cyclones. The primary cyclone underflow gravitates to the attritioning cells and then pumped to the secondary hydro-cyclones. The secondary cyclone underflow is washed on a stack of high frequency vibrating screens then dewatered on a dewatering screen and discharged to the rejects conveyor. The primary and secondary cyclone overflows and the stack sizer undersize is thickened in a paste thickener prior to being pumped to the HPAL autoclave.

Key criteria of the beneficiation section are shown in Table 17-2.

Table 17-2 Beneficiation operating parameters

Item	Unit	Value
Drum scrubber feed rate per train	dt/hr	341
Drum scrubber residence time	minutes	3
Drum scrubber pulp solids	%w/w	55
Drum scrubber discharge screen type	-	banana
Drum scrubber discharge screen top deck aperture	mm	30
Drum scrubber discharge screen bottom deck aperture	mm	6.3
Grit screen aperture	mm	3.5
Rejects wash screen aperture	mm	0.15
Primary cyclone overflow P_{80}	µm	40
Attrition cells residence time	minutes	4
Secondary cyclone overflow P_{80}	µm	50
Stack sizer high frequency screen aperture	µm	150
Consumed power	kW	2,683

17.3.2 High-pressure acid leach

The leaching of nickel and cobalt is completed through two HPAL trains, each consisting of a slurry feed tank, medium and high-pressure direct contact heater vessels, autoclave, three stages of flash cooling, associated pumps, piping, reagent and utilities. The autoclave is a six compartment, horizontal type, constructed of titanium GR17 clad steel, as commonly used on lateritic projects around the world. Heater vessels are also titanium GR17 clad steel and flash vessels are brick lined. Key design parameters for the leach are shown in Table 17-3 and Table 17-4.

Table 17-3 HPAL design parameters

Item	Unit	Value
Ore feed rate per train	dt/hr	134
Pulp solids concentration	%w/w	43
Feed tank surge capacity	hr	8
Autoclave residence time	minutes	60
Discharge free acid	g/ℓ	40
Acid consumption	kg/t	347
High pressure steam consumption	kg/t	187
Nickel extraction	%	97
Cobalt extraction	%	97
Consumed power	kW	2,280

Table 17-4 HPAL vessel design temperatures and pressures

Item	Temperature (°C)	Pressure (kPa abs)
Medium-pressure heater	144	400
High-pressure heater	192	1,300
HPAL autoclave	250	4,600
High pressure flash vessel	194	1,300
Medium pressure flash vessel	146	400
Low pressure flash vessel	103	101

Slurry from the leach feed tank is pumped directly into the first stage pre-heater where it is contacted with medium pressure flash steam from the autoclave discharge slurry letdown system. This heats the slurry and the system must be operated in a pressurised state to maximise energy recovery. Slurry from the first-stage heater is pumped to the second stage heater under increasing pressure, by two stages of centrifugal pumps. The second-stage preheater operates in a similar manner, where it is contacted with high pressure flash steam from the high pressure (first) flash vessel. The discharge slurry from the second-stage preheater is pumped into the autoclave by a piston diaphragm pump fitted with a thermal barrier leg to enable pumping at high temperature.

Concentrated sulphuric acid is sparged into the first two compartments of the autoclave via a tantalum lined titanium lance to provide the required acid dose for effective leaching of the nickel and cobalt. The reaction of the acid with water is exothermic and generates heat. The heat from the reaction of the ore with acid provides further heating. High-pressure steam is also sparged into compartments one and two of the autoclave to maintain and control the required operating temperature of 250°C. The residence time through the autoclave (60 minutes) is chosen to optimise nickel and cobalt leach extraction while also minimising net iron extraction.

The autoclave discharges to a three-stage flash circuit. The pressure is reduced through each stage via a ceramic choke valve and blast tube, which flashes steam and cools the slurry. Flashed steam from the first and second flash vessels (high and medium pressure) is ducted to the feed heaters, as previously discussed. Steam from the third and final flash vessel is ducted to the hot water heater in the beneficiation area. The final flash cooled leached slurry is pumped to neutralisation and pre-reduction.

17.3.3 Neutralisation and pre-reduction

Similar to HPAL, there are two trains of neutralisation and pre-reduction. Each train consists of four neutralisation tanks, two pre-reduction tanks, a large leach residue filtration surge tank and associated pumps, piping, reagent dosing and utilities.

The HPAL discharge slurry contains too much free acid for direct recovery of nickel and cobalt to work efficiently. A minor recycle stream, impurity removal filter cake, is added to the HPAL discharge slurry to re-leach nickel and cobalt contained in the cake. Limestone slurry is added in a series of four cascading agitated tanks to neutralise the residual free acid. The neutralisation reaction causes precipitation of gypsum[7] and precipitates most of the iron (III), aluminium, and chromium (III) as hydroxides. It also causes evolution of carbon dioxide gas, which is denser than air and poses an asphyxiation risk. The tanks are enclosed, and the vent is ducted to a stack. The pH is controlled to avoid co-precipitation of nickel and cobalt while removing most of the iron and some of the aluminium.

The key operating parameters for the neutralisation area are given in Table 17-5.

[7] Reports to tails

Table 17-5 Neutralisation design parameters

Item	Unit	Value
HPAL discharge solids rate per train	dt/hr	112
Pulp solids concentration	%w/w	33
Impurity removal solids addition per train	kg/hr	45
Operating temperature (approximate)	°C	80–95
Target discharge pH	pH	2.5–3.0
Limestone slurry solids concentration	%w/w	32
Limestone consumption (neutralisation and pre-reduction)	kg/t	225
Limestone utilisation	%	90
Number of tanks per train	no.	4
Residence time per tank	minutes	30
Consumed power	kW	156

Neutralised slurry overflows from the last neutralisation tank into a series of two cascading agitated tanks. Hydrogen sulphide gas is added to reduce the iron (III) and chromium (VI) to iron (II) and chromium (III) respectively. It also causes partial precipitation of copper as copper sulphide and generates acid from the reduction process, so limestone slurry is added to maintain the pH. The pre-reduction step eliminates the toxic and oxidising chromium (VI) and minimises impurities in the MSP intermediate product. Carbon dioxide and unused hydrogen sulphide gas in the pre-reduction tank vent is separated from the neutralisation tank vents and is directed to the MSP scrubber for hydrogen sulphide recovery.

The key operating parameters for the pre-reduction area are given in Table 17-6.

Table 17-6 Pre-reduction design parameters

Item	Unit	Value
Neutralised solids rate per train	dt/hr	151
Pulp solids concentration	%w/w	37
Operating temperature (approximate)	°C	80
Oxidation reduction potential	mV (Ag/AgCl)	<250
Hydrogen sulphide consumption per train	kg/hr	8
Hydrogen sulphide utilisation	%	95
Number of tanks per train	no.	2
Residence time per tank	minutes	30
Consumed power	kW	1,031

17.3.4 Leach residue filtration

The leach residue filtration area is divided into two trains to match the upstream HPAL, neutralisation and pre-reduction areas. Each train of the leach residue filtration area consists of a surge tank, three stages of pressure filtration, residue conveying and stacking, PLS pond, PLS polishing filter and associated pumps, piping and utilities. Each stage of residue pressure filtration includes a filter feed/re-pulp and filtrate tanks, feed and filtrate pumps, pressure filter, cake belt feeder, transfer conveyor and associated utilities.

The neutralised and reduced leach residue is filtered in three stages with counter-current re-pulp washing using recycled process water. Re-pulp washing has been specified due to the low cake porosity typical of nickel laterite HPAL residues, which causes long wash cycle times and poor wash recoveries. Three filters have been specified per stage per train and an equipment design factor of 1.2 has been used. Hence, plant throughput is restricted to 80% with two of the three filters online or capable of 120% with all three online. The leach residue surge tank, process water and PLS ponds provide surge to decouple the leach residue filtration area from upstream and downstream unit operations. Filter sizes have been standardised to minimise spares inventory, but additional plates have been added for the Stage 3 filter which includes the acid liquor neutralisation solids.

Key operating parameters for the leach residue filtration area are given in Table 17-7.

Table 17-7 Leach residue filtration design parameters

Item	Unit	Value
Leach residue surge tank capacity per train	hr	8
Pulp solids concentration	%w/w	37
Operating temperature (approximate)	°C	≤80
Number of filter stages per train	no.	3
Number of filters per stage per train	no.	3
Leach residue filter feed rate per train (Stages 1 and 2)	dt/hr	151
Leach residue filter feed rate per train (Stage 3)	dt/hr	196
Specific filtration rate	kg/(m².hr)	100
Filter availability	%	85
Wash ratio (process water to Stage 3)	t water / t solids	1.5
Cake moisture	%w/w	32
Filtrate solids	ppm	100
Filter feed/re-pulp tank residence time	hr	1
Filtrate tank residence time	hr	1
PLS pond residence time	hr	24
Consumed power (includes PLS pond and polishing filters)	kW	1,093

The final (Stage 3) cake is transferred by belt feeder and conveyors to the residue disposal area for dry stacking. Stage 1 filtrate is collected and pumped to the PLS pond. PLS is pumped from the PLS pond to the MSP area via the PLS polishing (pressure candle type) filters along with associated backwash tank, pumps and piping.

PLS parameters and composition are given in Table 17-8.

Table 17-8 PLS design parameters

Item	Unit	Value
PLS pond residence time	hr	24
Nickel	g/ℓ	7.24
Cobalt	g/ℓ	0.38
Copper	mg/ℓ	36.1
Zinc	mg/ℓ	480
Magnesium	g/ℓ	42.9
Manganese	g/ℓ	1.65
Iron	g/ℓ	1.85
Aluminium	mg/ℓ	73.2
Calcium	mg/ℓ	602

17.3.5 Mixed sulphide precipitation

As per the upstream areas, the MSP area is divided into two trains. Each MSP train consists of three agitated reactor vessels, a flash letdown vessel, flash recovery compressor, thickener, filter feed tank, pressure filter, overflow tank, polishing filter, filtrate and backwash tanks, recycles tank, vent gas scrubber, associated pumps, piping and utilities.

The filtered PLS contains soluble nickel and cobalt sulphates plus some other gangue metal cations (zinc, magnesium, manganese, and minor iron, aluminium and calcium, etc.). The solution is pumped to a series of precipitation reactor vessels where nickel and cobalt is precipitated by direct sparging of hydrogen sulphide gas. Direct injection of low-pressure steam is used for heating and recycle surge tank and pumping is included to recover various refinery and scrubber bleed streams to the reactor vessels.

The mixed base metal sulphide (nickel sulphide and cobalt sulphide) exits the pressurised reactor vessels via a flash vessel and is then thickened. Flash vapour is re-compressed and injected back into the reactor vessels to maximise hydrogen sulphide utilisation. A large recycle of solids in the thickener underflow is used to act as a seed and promote particle growth in the reactor vessels.

Key MSP parameters are given in Table 17-9.

Table 17-9 MSP design parameters

Item	Unit	Value
PLS flow per train	m^3/hr	326
PLS density	t/m^3	1.21
Recycles flow per train	m^3/hr	10
Recycles density	t/m^3	1.08
Number of MSP tanks per train	no.	3
MSP reactor residence time per vessel	minutes	10
Operating temperature	°C	95
Operating pressure	kPa (gauge)	200
Hydrogen sulphide consumption per train	t/hr	1.5
Hydrogen sulphide utilisation (excluding scrubber recycle)	%	95
Nickel recovery	%	99
Cobalt recovery	%	99
Low pressure steam consumption per train	t/hr	11.9
Caustic soda (100 % basis) consumption per train	kg/hr	187
Consumed power (includes thickening and filtration)	kW	423

The remaining thickener underflow slurry is filtered and washed with demineralised water in a pressure filter, to produce the MSP. The filtrate drains by gravity into the thickener overflow tank which is pumped to polishing filters to minimise the loss of solids to the barren liquor. Allowance has been made for bagging the mixed sulphide for added surge capacity between MSP and the downstream processing. MSP thickening and filtration parameters are shown in Table 17-10.

Sulphide circuit vapours are scrubbed in a gas scrubber, with sodium hydroxide solution addition. The scrubbing reaction generates sodium hydrosulphide solution, which is returned to the sulphide precipitation reactors, to improve the net sulphide utilisation.

Table 17-10 Mixed sulphide thickening and filtration design parameters

Item	Unit	Value
Thickener underflow solids concentration	%w/w	40
Seed recycle rate	%	400
Flocculant consumption	g/t	60
Specific settling rate	kg/(m².hr)	300
Filter feed tank residence time	hr	8
Filter cake moisture	%	30
Filter cake wash ratio	t water / t solids	2
Filter wash efficiency	%	99
Specific filtration rate	kg/(m².hr)	150

17.3.6 Pressure oxidation leach

Unlike the upstream processes, the POX area consists of a single train that processes all mixed sulphide produced (i.e. from both MSP trains). The mixed nickel-cobalt sulphide filter cake is re-pulped with demineralised water in a re-pulp tank in each train, then pumped to the single MSP surge tank then autoclave feed tank. The mixed sulphide slurry is pumped by piston diaphragm pump to a five-compartment horizontal autoclave with the key design parameters shown in Table 17-11.

Oxygen is sparged along the length of the vessel to fully oxidise the sulphides to sulphate and solubilise nickel and cobalt. Slurry is withdrawn from compartment one or two and flash cooled to remove heat that is generated from the sulphide oxidation process before being recycled back to the autoclave feed tank. The amount of water fed to the autoclave with the sulphides is adjusted to maintain a target discharge nickel concentration. The autoclave discharges from compartment five via an atmospheric flash vessel and is pumped to the impurity removal circuit. Additional cooling is achieved by recycling some of the autoclave discharge liquor back to the autoclave via a fin fan cooler. The autoclave vents are directed to a barometric condenser and scrubber.

Table 17-11 POX leach design parameters

Item	Unit	Value
MSP surge tank residence time	hr	8
MSP feed rate	dt/hr	7.9
Pulp solids concentration	%w/w	10
Autoclave operating temperature	°C	165
Autoclave operating pressure	kPa (gauge)	1,700
Autoclave residence time	hr	3.4
Oxygen consumption	t/hr	
Oxygen utilisation	%	80
Discharge nickel concentration	g/ℓ	100
Scrubber caustic soda (100% basis)	kg/hr	1.0
Consumed power	kW	1,544

17.3.7 Impurity removal

The purpose of the impurity removal area is to adjust the pH of the PLS from the POX discharge and to precipitate the remaining trace amounts of copper, iron, aluminium and chromium from solution. As per the POX area, the impurity removal circuit consists of a single train. The impurity removal circuit includes six reactor tanks followed by a clarifier, overflow tank, polishing filter and backwash tank and pumps, clarifier underflow recycle and advance pumps, filter feed tank and pressure filter. The impurity removal filter cake is recycled back to HPAL discharge (neutralisation area) to leach any co-precipitated nickel and cobalt.

The key operating parameters for the impurity removal circuit are given in Table 17-12.

Table 17-12 Impurity removal design parameters

Item	Unit	Value
Impurity removal feed flow	m^3/hr	46.9
Number of tanks	no.	6
Residence time per tank	minutes	30
Final pH	pH	5.5
Caustic soda (100 % basis) consumption	t/hr	2.38
Consumed power	kW	33

A portion of the cobalt solvent extraction raffinate and cobalt sulphate crystalliser bleed is contacted with sodium hydroxide solution to precipitate nickel and cobalt as hydroxides and hydroxy-sulphates. The pH is controlled to precipitate all the nickel and cobalt, but the barren liquor is returned to the MSP recycles surge tank to recover nickel and cobalt during any process upsets. The slurry is filtered in a small pressure filter and washed well in demineralised water to displace any sodium entrained in the cake moisture. The filter cake is re-pulped in demineralised water and dosed into the impurity removal reactors, avoiding the addition of sodium to the PLS, which would otherwise concentrate in the nickel sulphate crystalliser and contaminate the nickel sulphate final product.

17.3.8 Cobalt solvent extraction

Consistent with POX and impurity removal areas, the CoSX area consists of a single train. The purified PLS from the impurity removal polishing filter is pumped into the CoSX circuit which consists of extraction, scrubbing, sequential cobalt and zinc stripping and nickel pre-loading stages. The circuit uses industry standard Cyanex 272 extractant in a low aromatic content, high flashpoint diluent, to extract the cobalt, leaving the nickel in the raffinate.

Magnesium and calcium impurities are rejected via spent scrub solution to the mixed nickel and cobalt hydroxide precipitation circuit (refer to impurity removal description). Manganese, copper and calcium impurities report to the cobalt loaded strip solution where ion exchange is used to remove these impurities from the cobalt-rich solution. Zinc and iron are stripped separately after the cobalt. Nickel pre-loading is used to avoid pH control in the extraction stages and separate sodium from the nickel and cobalt product streams. Dual media filter and carbon columns are used to control organic content in the product streams.

Key CoSX design parameters are given in Table 17-13.

Table 17-13 CoSX design parameters

Item	Unit	Value	O/A ratio*	pH
PLS flow	m^3/hr	48	-	5.5
Cyanex 272 concentration	%v/v	20	-	-
Extraction stages	no.	4	0.675	4.4–5.6
Scrub stages	no.	5	4.0	3.9–4.2
Cobalt strip stages	no.	3	9.6	2.5
Zinc strip stages	no.	1	20	0.2–0.5
Nickel pre-load stages	no.	2	2.85	5.7–6.5
Sulphuric acid (98% basis) consumption	t/hr	1.06	-	-
Caustic soda (100% basis) consumption	t/hr	6.77	-	-
Consumed power	kW	60	-	-

**O/A ratio – organic to aqueous phases volume ratio.*

17.3.9 Nickel and cobalt crystallisation

The cleaned cobalt loaded strip liquor at pH 3 contains 80 g/ℓ cobalt as sulphate, plus only approximately 300 ppm free H_2SO_4. The liquor is evaporated in a single draft tube baffle crystalliser, operating under vacuum at approximately 50 mBar absolute pressure and less than 40°C. The vacuum is generated by steam-jet thermo vapour re-compression (TVR). A portion of the overhead vapours is re-compressed by the motive steam. This reheats the vapours and allows them to act as heating medium in the crystalliser heat exchanger, thus reducing the net steam demand. The remaining process vapours are re-compressed with a small vacuum booster pump, to raise the pressure and temperature and allow the vapours to be condensed by cooling water in a plate condenser, without needing chilled water.

The crystalliser includes a forced circulation leg, which draws liquor from near the surface of the vapour liquid interface by a low head, high flow axial flow pump. The re-circulating liquor is heated in a shell and tube heat exchanger to just under the boiling temperature, under hydrostatic pressure only. The heated liquor returns to the crystalliser vessel where it flashes. The cobalt sulphate crystallises as the heptahydrate: $CoSO_4 \bullet 7H_2O$, which is the stable phase at the operating temperature and pressure. Settled slurry is pumped to a pusher centrifuge, with a portion of the centrate bled to mixed nickel and cobalt hydroxide precipitation, and the remainder returned to the crystalliser. The cobalt sulphate is centrifuged, and dried, before being packaged in bulk bags for product storage and distribution.

The CoSX raffinate contains concentrated nickel sulphate, which is then pumped to the nickel sulphate crystalliser. The circuit includes two stages, with a first evaporation stage to reach saturation by mechanical vapour re-compression (MVR) and a second crystallisation stage. The hexahydrate nickel sulphate is produced: $NiSO_4 \bullet 6H_2O$, which is the stable phase at the operating temperature and pressure which is slightly higher pressure than the cobalt sulphate crystalliser. The nickel sulphate is centrifuged and dried, before being transferred to lined shipping containers for product storage and distribution. As with the cobalt sulphate crystal product, some of the centrate liquid from the nickel sulphate product is bled to the nickel pre-load cell within the CoSX circuit to control the build-up of impurities in the nickel sulphate mother liquor.

Both the nickel and cobalt sulphate products will be produced to a high purity suitable for the battery market. The process flowsheet selected including MSP, POX, impurity removal (metal hydroxide precipitation), CoSX and ion exchange on the cobalt loaded strip liquor will remove impurities to low levels suitable for the battery products market.

17.3.10 Acid liquor neutralisation

The acid liquor neutralisation area neutralises plant effluent and recycles it as process water. There are two trains of acid liquor neutralisation, consistent with the upstream MSP area. Barren liquor from the MSP area is the main input to the acid liquor neutralisation area but minor streams such as zinc strip liquor, POX scrubber bleed and the hydrogen electrolyser scrubber bleed also feed into the acid liquor neutralisation area. The area consists of two agitated neutralisation tanks, air blower and receiver, thickener, overflow tank, process water pond and associated pumps, piping, reagents and utilities.

Limestone slurry is added to the effluent streams in the first tank to neutralise any free acid. The pH is increased further in the second tank by the addition of slaked lime. Air is blown through both tanks to oxidise any residual sulphide, convert iron (II) to iron (III) and oxidise manganese (II) to manganese (IV) facilitating iron and manganese precipitation. At this pH, most other metal cations are also precipitated with the exception of magnesium. The slurry is then thickened, the overflow is pumped to the process water pond and the underflow is split with a seed recycle pumped back to the neutralisation tanks and the remaining underflow pumped to the third stage of leach residue filtration. The key operating parameters for the acid liquor neutralisation area are given below in Table 17-14.

Table 17-14 Acid liquor neutralisation design parameters

Item	Unit	Value
Effluent inflow per train	m^3/hr	367
Effluent liquor density	t/m^3	1.17
Number of neutralisation tanks	no.	2
Residence time per tank	minutes	30
Operating temperature	°C	80–90
Tank 1 pH	pH	4.5–5.0
Tank 2 pH	pH	9.0–9.5
Air consumption	Nm^3/hr	21,677
Oxygen utilisation	%	10
Limestone consumption	dt/hr	19.5
Slaked lime consumption	dt/hr	22.6
Thickener specific settling rate	t/(m^2.hr)	0.4
Underflow solids concentration	%w/w	37
Flocculant consumption	g/t	200
Seed recycle rate	%	400
Consumed power	kW	1,556

17.3.11 Kieserite (magnesium sulphate) crystallisation

To avoid accumulation of magnesium sulphate in the recycled process water, a portion of the process water is sent to the kieserite crystallisation area. Similar to the acid liquor neutralisation area, there are two trains of kieserite crystallisation. Waste steam from the HPAL area is used in the kieserite crystalliser.

The feed liquor is first concentrated in a falling film pre-concentrator which is operated by means of MVR units. Pre-concentrator operates close to the solubility limit before feeding to the crystalliser. The crystalliser is a forced circulation type of device, where the solution concentrates at the same time when the crystals are formed. The crystalliser is operated by TVR principle (i.e. vapor is compressed by means of ejector back to heater). Excess amount of vapor is condensed in surface condenser.

Crystal slurry from the crystallisation vessel is pumped to the thickener and thickened slurry is led to the centrifuge for solid-liquid separation. The separated crystals are then dried in a fluid bed dryer to final dryness. Drier off-gas is purified by means of cyclones and scrubber. Mother liquor from the thickener and from centrifuge is collected into the mother liquor tank and most of it is recycled back to crystallisation.

Kieserite crystallisation parameters are shown in Table 17-15.

Table 17-15 Kieserite crystallisation design parameters

Item	Unit	Value
Process water feed per train	t/hr	45.2
$MgSO_4$ concentration	%w/w	15.3
Operating temperature	°C	87
Kieserite ($MgSO_4 \cdot H_2O$) production per train	t/hr	7.8
Hot water heater steam	t/hr	38.9
Consumed power	kW	509

17.3.12 Sulphuric acid production

Dual trains of 1,200 t/d acid plants have been included in the plant design. This is consistent with the dual trains of HPAL and associated processing plant. Outotec sulphur burning, double absorption, acid plants equipped with HEROS™ heat recovery have been selected. The cost compares favourably with that of a water-cooled design, and the additional low-pressure steam and the extra power produced compared to conventional acid plant designs assists with the process plant heat balance and kieserite crystallisation area steam requirements.

The acid plant consists of the following main components:

- Sulphur melting and filtration
- Sulphur combustion
- Waste heat recovery system
- Strong acid system
- HEROS™ heat recovery technology
- Converter system.

17.3.13 Reagent handling

The following reagents are used in the process:

- Sulphur – Sulphur is imported as bulk flake or prill for on-site production of sulphuric acid in a sulphuric acid plant.
- Oxygen – Liquefied oxygen is produced and stored on site by a cryogenic oxygen plant.
- Limestone – Locally sourced limestone is crushed (maximum top size 300 mm) and milled in process water in closed circuit with hydro-cyclones to produce a limestone slurry for distribution to the process. The limestone slurry has 32%w/w solids concentration and a particle size of 80% passing 150 µm.
- Caustic soda – Caustic soda (sodium hydroxide) is delivered as 50% w/w solution. It is diluted to 10% w/w and distributed to solvent extraction and off-gas scrubbers throughout the process.
- Hydrogen – Hydrogen is produced on site from the water hydrolysis plant. The hydrogen is fed to the hydrogen sulphide plant and is used solely for the production of hydrogen sulphide gas. Oxygen is a by-product from the water electrolysis plant and is used to supplement the oxygen produced in the cryogenic plant.
- Hydrogen sulphide – Hydrogen sulphide gas is produced on site in a hydrogen sulphide plant, in which hydrogen is injected into molten sulphur under pressure to produce hydrogen sulphide gas.
- Quicklime – Quicklime is slaked on site to produce a milk of lime (hydrated lime) slurry for distribution to the process. The milk of lime slurry has a solids concentration of 20% w/w.

- Minor reagents – Minor reagents such as selective organic extractants, ion exchange reagents, flocculants and water treatment chemicals are stored on site in appropriate containment and prepared daily as needed.

17.3.14 Utilities

Water systems

The plant water system consists of the following components:

- Raw water pond
- Fire water system
- Water treatment plant (filtration and sterilisation)
- Potable water storage and distribution
- Demineralised water plant and distribution system
- Process water pond distribution system
- Cooling water system.

A portion of the raw water is pumped to the water treatment plant. Due to the high quality of local water, the water treatment plant consists of filtration and sterilisation only. It produces potable water for human consumption and for feed to the demineralised water plant. Potable water is distributed via a header pipe system and pressure control pump.

Demineralised water is stored in a single storage tank for distribution. The primary use of demineralised water is boiler feed water makeup in the sulphuric acid plant. Demineralised water is also used in the water electrolyser plant to generate hydrogen and to re-pulp the mixed sulphide filter cake as feed to the POX leach autoclave and for reagent make-up where required. Demineralised water is distributed via a header pipe system and pressure control pump.

Process water is recycled within the plant and topped up with raw water as required. The process water is mainly consumed in the beneficiation area via the hot water heater and as wash water in the leach residue filtration area. Process water is distributed via a header pipe system and pressure control pump.

Compressed air

The compressed air system consists of the following components:

- Plant air compressor (700 Nm^3/hr at 850 kPa absolute)
- Plant air receiver and distribution header system
- Filter air compressor (755 Nm^3/hr at 850 kPa absolute)
- Filter air receiver and distribution piping
- Instrument air dryer (80 Nm^3/hr at 850 kPa absolute)
- Instrument air receiver and distribution header system.

The plant air system feeds the instrument air dryer and filter air system is separated from normal plant air to ensure filters have sufficient air at all times.

Backup low pressure steam boiler

A single diesel fired, low pressure steam boiler (1 t/hr steam at 1,000 kPa gauge) is provided to enable acid plant start-up (i.e. sulphur melting and heat tracing etc. only). Only one backup boiler is required. Once the first acid plant is running, steam is available to start the second acid plant.

Diesel storage

Provision has been made for a diesel storage vessel, distribution pump and associated piping (connected to the backup low pressure steam boiler; and fuel bower for light vehicle use).

Steam system and turbine generator

A steam turbine generator is installed for each acid plant train. The model Siemens SST300 which is capable of up to 50 MW of power generation has been selected. It is a single-casing steam turbine, providing geared drive to 1,500 or 1,800 rpm generator, and is often used in process plants using waste heat for power generation.

Electrical power

Electrical power is consumed in each area and the below table shows the proportion of electrical consumption by these respective areas. Details are also shown in Table 17-16 for on-site produced power and nominal electrical power required from the grid.

Table 17-16 Power consumption by work breakdown structure area (kW)

Work breakdown structure (WBS)	WBS no.	Consumed power
Crushing	110	670
Drum scrubbing	120	1,053
Attrition scrubbing and screening	130	927
Leach feed thickening	140	704
HPAL	210	2,280
Slurry neutralisation and leach residue filtration	220	5,442
MSP	420	423
MSP POX	430	1,544
Impurity removal	435	33
Co Zn solvent extraction	440	60
$CoSO_4$ crystallisation	450	77
$NiSO_4$ crystallisation	460	932
Acid liquor neutralisation	510	1,557
Kieserite crystallisation	530	509
Sulphuric acid plant	610	5,451
Limestone	620	723
Slaked lime	630	345
Flocculant	650	14
Sodium hydroxide	670	4
Hydrogen sulphide	681	559
Water electrolysers[8]	682	9,657
Oxygen plant	690	2,202
Raw water	710	49
Water treatment	720	696
Cooling water	730	1,409
Demineralised water	740	15
Steam turbine generator	810	511
Plant air	820	142
Total consumed power		**37,988**
Available power from site generation		**22,932**
Power required from grid		**15,056**

[8] Used to generate H_2 for H_2S gas production which is used to precipitate the nickel and cobalt as mixed sulphide precipitate MSP

17.4 Mass and energy balances

The process mass balances have been developed using SysCAD software. The HZM SysCAD model was developed by Simulus. The information in these balances conforms to the level of accuracy required for this PFS. The mass balance has been run at 85% availability or 7,446 hours per year.

The design basis used is year 9 of the mine plan with a nominal ROM ore of 4.34 Mt/a at an average grade of 1.07% Ni which is upgraded (by the proposed beneficiation plant) to 2 Mt/a at an average grade of 1.85%. The minerals in the ore were developed to be self-consistent with the ore element chemical analysis and was provided by HZM.

The mass and nickel balance from the SysCAD model for the Project is summarised in Table 17-17, and mass balance for reagents and water in Table 17-18.

Table 17-17 Mass balance nickel and cobalt summary

Description	Stream	Units	Total solids	Nickel		Cobalt	
				kg/hr	%	kg/hr	%
ROM solids	110-001	kg/hr	582,710	6,260	100	371	100
Beneficiation rejects	120-008	kg/hr	314,114	1,299	20.8	108	29.0
Beneficiation concentrate	130-014S	kg/hr	268,601	4,961	79.2	264	71.0
Combined plant tailings	220-014	kg/hr	391,467	276	5.6*	13	5.1*
Cobalt sulphate product	450-024	kg/hr	1,193	0	0.00*	250	94.9*
Nickel sulphate product	460-024	kg/hr	20,981	4,685	94.4*	0	0.0*
Kieserite crystals	530-020	kg/hr	15,630	0	0.0*	0	0.0*
Ball mill trommel oversize	620-004	kg/hr	306	0	0.0*	0	0.0*

**Metal recovery normalised for HPAL feed (i.e. excludes beneficiation rejects).*

Table 17-18 Mass balance reagents and water summary

Reagents and water list	Stream	Mass rate (t/hr)	Volume rate (m^3/hr)	Feed rate by weight (kg/t HPAL feed)	Feed rate in by volume (ℓ/t HPAL feed)
Cyanex 272	440-001	0.002	0.002	0.007	0.007
Diesel	830-011	0.0001	0.0001	0.0004	0.0004
Diluent	P_296	0.006	0.008	0.024	0.030
Flocculant	650-100	0.100	0.134	0.373	0.498
Lime	630-001	17.178	5.172	63.954	19.257
Limestone	620-001	73.820	29.908	274.831	111.348
NaOH	670-001	2.581	1.696	9.609	6.316
Oxygen	P_088	5.354	4,096.349	19.933	15,250.709
Sulphur	610-007	33.604	16.234	125.106	60.438
Demineralised water	720-006a	125.965	126.338	468.967	470.355
Raw water	710-001	125.112	125.463	465.791	467.099

A complete list of all emissions and effluents from the processing plant are given in Table 17-19. The process has been designed with the principle of zero liquid discharge. Therefore, there are only two waste solids streams, the coarse beneficiation rejects and the washed leach residue filter cake. There is no liquid effluent. Products and by-products (nickel and cobalt sulphate and kieserite crystals) are not included in this list. SysCAD process simulation model headers are shown in Table 17-20.

Table 17-19 Effluents and emissions

Item	Stream	Units	Value	% solids
Beneficiation coarse rejects	120-014	dt/hr	314	86.1
Washed leach residue solids	220-020	dt/hr	391	68.0
Slurry neutralisation vent	220-011	Nm^3/hr	15,425	-
MSP scrubber vent	420-020	Nm^3/hr	603	-
POX scrubber vent	430-029	Nm^3/hr	1,438	-
Cobalt crystalliser vent	450-013	Nm^3/hr	<5	-
Cobalt product dryer/scrubber vent	450-022	Nm^3/hr	1,325	-
Nickel crystalliser vent	460-013	Nm^3/hr	<5	-
Nickel product dryer/scrubber vent	460-023	Nm^3/hr	23,067	-
Acid liquor neutralisation tank vents	510-002	Nm^3/hr	49,592	-
Kieserite ($MgSO_4 \cdot H_2O$) crystalliser vent	530-013	Nm^3/hr	<5	-
Sulphuric acid plant tail gas	610-014	Nm^3/hr	152,901	-
Hydrogen sulphide plant tail gas	681-002	Nm^3/hr	200	-
Oxygen plant vent	690-002	Nm^3/hr	14,476	-
Steam turbine generator vent	810-028	Nm^3/hr	22,338,671	-
Deaerator blowdown vent	810-004	Nm^3/hr	741	-
Backup diesel boiler exhaust	810-008	Nm^3/hr	<5	-
Backup diesel generator exhaust	840-002	Nm^3/hr	<5	-

Table 17-20 SysCAD process simulation model headers

Model headers, 1-23	Model headers, 24-46
110-001 Crushing	460-002 $NiSO_4$ drying and packaging
110-002 Stockpiling and reclaim	510-001 Acid liquor neutralization
120-001 Drum scrubbing	530-001 Kieserite crystallization
120-002 Rejects handling	610-001 Sulphuric acid plant
130-001 Attrition scrubbing and screening	610-001 Sulphuric acid storage
140-001 Leach feed thickening	620-001 Limestone
210-001 HPAL	630-001 Slaked lime
220-001 Slurry neutralization	650-001 Flocculant
220-002 Leach residue filtration	670-001 Sodium hydroxide
220-003 Leach reside conveying	681-001 Hydrogen sulphide
420-001 Mixed sulphide precipitation	682-001 Water electrolysers
420-002 Sulphide precipitation scrubbing	690-001 Oxygen plant
430-001 MSP POX	710-001 Raw water
430-002 MSP POX vent gas scrubbing	720-001 Water treatment
435-001 Impurity removal	730-001 Cooling water
440-004 Co Zn SX extract	740-001 Demineralised water
440-005 Co Zn SX nickel scrub	810-001 Steam turbine generator
440-006 Co Zn SX cobalt strip	810-001 Backup boiler
440-007 Co Zn SX zinc strip	810-003 Steam distribution
440-008 Co Zn SX nickel preload	820-001 Plant air
450-001 $CoSO_4$ crystallization	820-002 Filter air
450-002 $CoSO_4$ drying and packaging	820-003 Instrument air
460-001 $NiSO_4$ crystallization	830-001 Fuel storage

18 PROJECT INFRASTRUCTURE

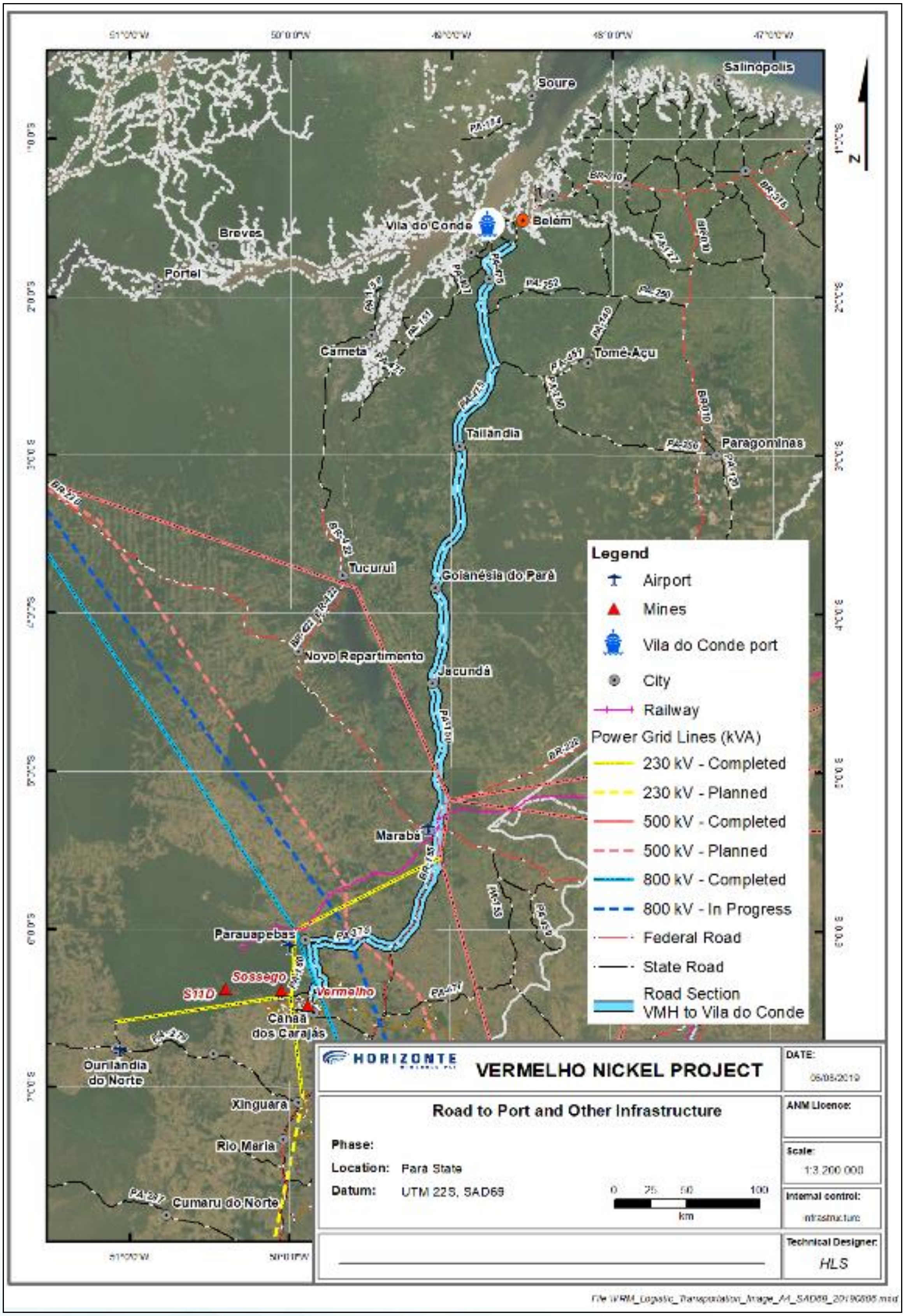
18.1 Summary

The scope of infrastructure described within this section includes the Project site requirements as well as the existing road infrastructure, identified as the transport route to the Port at Vila do Conde near Belém in the state of Pará for the import of sulphur for the Project, and the export of nickel and cobalt sulphate product (Figure 18-1).

The proposed infrastructure for the Project will include:

- Access and site roads
- Water supply, water treatment and mine site sewage treatment
- Power supply
- Beneficiation and process residue storage facility
- Security and fencing
- Fire-fighting system
- Administration and maintenance buildings, including laboratory
- Data and communications infrastructure.

Figure 18-1 Road to port and other infrastructure



Source: HZM, 2019

18.2 Logistics solution for Vermelho

A comprehensive logistics study was completed by C. Steinweg Handelsveem (Latin America) S.A. (Steinweg) for HZM. The study aimed to identify the most cost-effective method to transport the following items:

- Sulphur from the selected port of entry to the plant site
- Limestone to the plant site (sourced within Brazil)
- Nickel and cobalt sulphate product from plant site to a port of export
- Other reagents and consumables to the Project.

Product shipment and inbound sulphur will be via Vila do Conde. Vila do Conde is located in Barcarena (PA) on the right bank of the Pará River, at Ponta Grossa, on the confluence of the Amazon, Tocantins, Guamá and Capim rivers. The wharf is constructed in a “T” shape with four berthing cradles: one for dry bulk materials, one for general cargo, one for caustic soda and one for liquid materials. The berths are aligned with the flow of the Pará River and they are connected to the land by a bridge access 378 m in length.

The port is approximately 687 km from the Project site by road (Table 18-1).

Table 18-1 Road segments, site to port

Segment name	From	To	Distance (km)	Responsibility
HZM access road	Site	PA160 intersection	11	HZM
PA160	P160 Intersection	Parauapebas	45	Pará State
PA275	Parauapebas	Eldorado dos Carajás	69	Pará State
BA 150/155	Eldorado dos Carajás	Goianésia do Pará,	274	Brazil National
PA 476	Goianésia do Pará,	Barcarena	288	Pará State
Total			**687**	

A summary of the logistics solution is provided in Table 18-2. Steinweg estimated transport costs of these key reagents are detailed in Section 21.

Table 18-2 Logistics solution

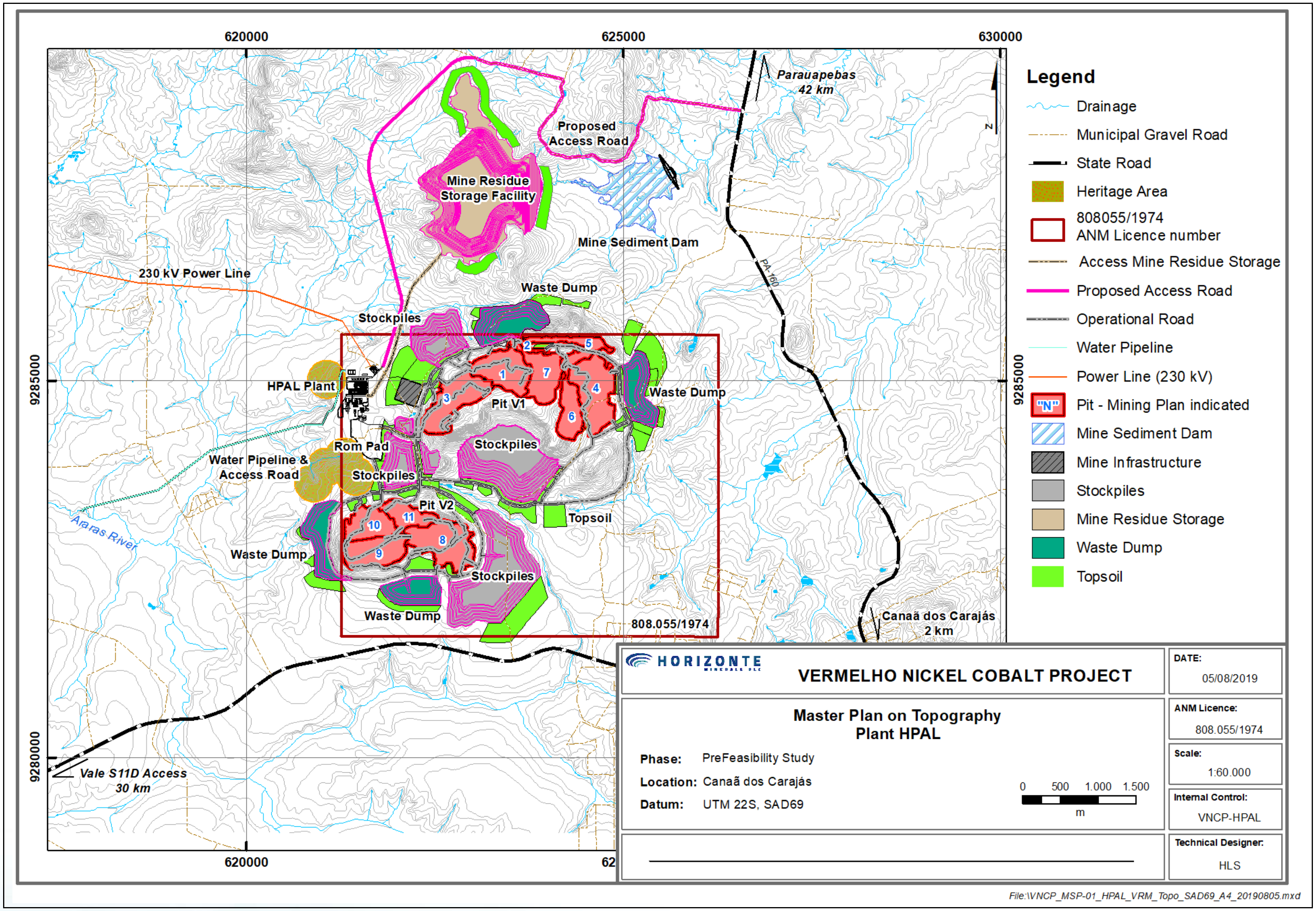
Item	Typical annual quantity (t)	Direction	Logistics solution
Sulphur	250,000	Inbound	Shipped in bulk from Vancouver to Vila do Conde in Barcarena (PA) then loaded onto 25-t trucks from the ship to be transported to the site
Limestone	393,000	Inbound	Trucked to site in 25-t trucks from Xambioá (TO)
Nickel sulphate	156,000	Outbound	Bagged at the plant, placed in containers, and then trucked to Vila do Conde in Barcarena (PA) where it is shipped to the customer (assumed Shanghai)
Cobalt sulphate	9,000	Outbound	
Lime	128,000	Inbound	Trucked in big bags from Taboão da Serra (SP)
Other reagents	Various	Inbound	By truck from suppliers in Brazil

Source: Steinweg, 2019

18.2.1 Roadworks for the Project

Access to the plant site and the main offices will be via new 11 km road that is constructed from State Highway PA169 to the plant site (Figure 18-2). An engineering cost estimate covering the construction and maintenance of this access road was completed by Construserv Servicos e Construcoes Ltda (CST). This estimate is detailed in Section 21.

Figure 18-2 Vermelho site layout



Source: HZM, 2019

18.3 Water supply

WALM was commissioned by HZM to develop an Integrated Water Balance for the Project. Their findings are summarised below in Table 18-3. The Project water consumption is estimated to be 316 m³/hr.

Table 18-3 Plant water consumption

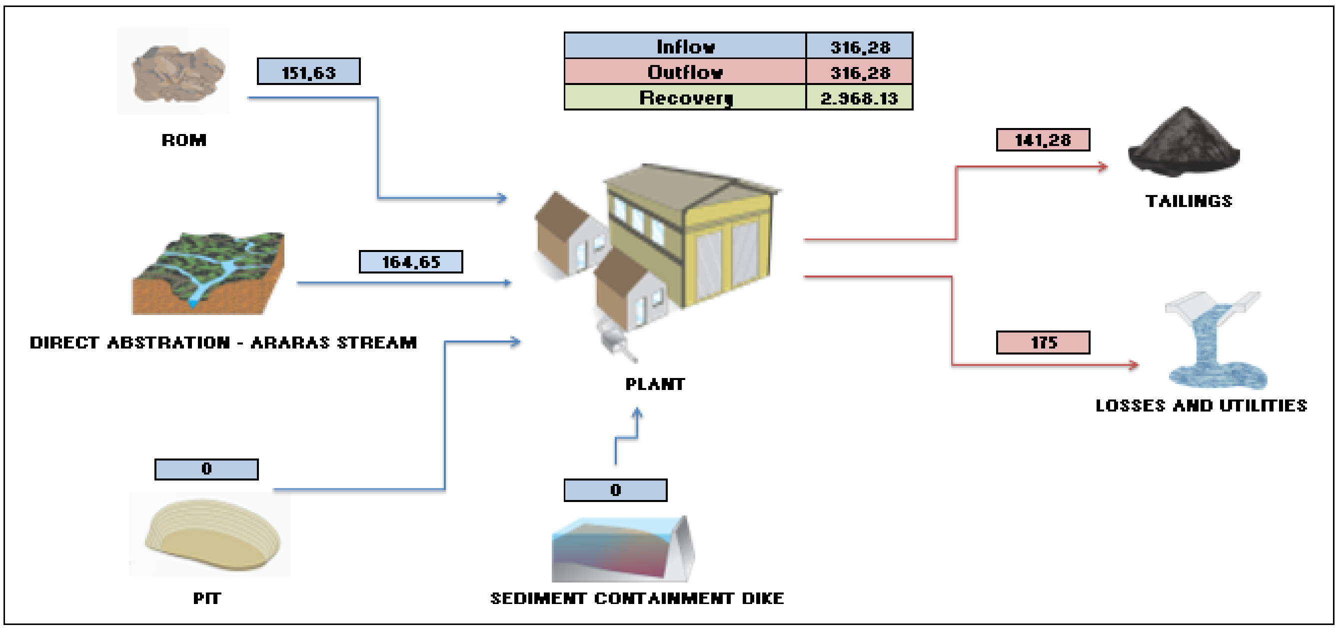
Variables	Uses (m³/hr)	Recovery rate (m³/hr)	Recovery rate (%)	Losses (%)	Losses (m³/hr)
Water for air cooling	14.74	0	0 %	100%	14.74
Raw water	129.88	0	0%	100%	129.88
NaOH mixing tank	10.26	0	0%	100%	10.26
Flocculation mix	20.02	0	0%	100%	20.02
Water electrolysis	0.10	0	0%	100%	0.10
Cooling tower	2,968	2,968	100%	0%	0
Subtotal	**3,143**	**2,968**	**-**	**-**	**175**
Water in the tailings and rejects	141.00	0	0%	100%	141
Total	**3,284**	**2,968**	**90.4%**	**9.6%**	**316**

Source: WALM, 2019

The main makeup water source will be a run-of-the-river direct abstraction from the right margin of the Araras stream (UTM 617,826.85E e 9,283,271.53S,) about 2.5 km from the Project site (Figure 18-2).

The water balance that WALM developed is shown in Figure 18-3 below.

Figure 18-3 Water balance schematic



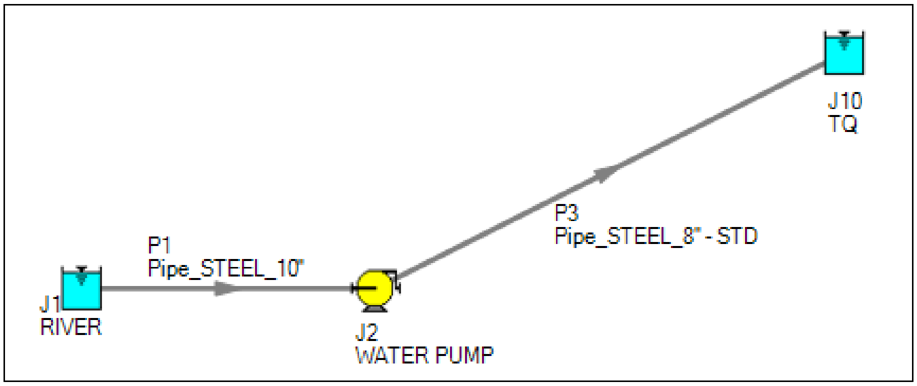
Source: WALM, 2019

WALM determined that the makeup water from the river is 165 m³/hr, and the remainder will come from the inherent moisture in the ROM (estimated at approximately 20%).

HZM commissioned Miptec EngEnharia Consultoria (Miptec) to design the river abstraction barge, pipes and pumping system for the Project. A summary of Miptec's findings is shown below.

Miptec developed a 4.5 km route from the abstraction point to the plant raw water tank (Figure 18-2). A schematic of the system and details are shown in Figure 18-4 and Table 18.4.

Figure 18-4 River abstraction schematic



Source: MIPTEC, 2019

Table 18-4 River abstraction pipework

Item	Type	Design flow (m^3/s)	Velocity (m/s)	Inlet level (m)	Outlet level (m)	Length (m)
P1	10" Steel	204	1.114	210	210	12
P3	12" Steel	204	1.756	210	277	4,500

Miptec developed detailed engineering drawings, selected and undertook a capital cost and operating cost estimate. These are detailed in Section 21.

18.4 Mine residue storage facility

HZM commissioned WALM to develop a scheme whereby the dry-stacked tailing from the HPAL plant and the beneficiated rejects could be safely co-disposed in a mine residue storage facility.

The physical properties and expected quantities of the various components of the facility are presented in Table 18-5.

Table 18-5 Dry stacked tailings and rejects key physicals

Parameter	Unit	Mine residue component			
		Primary	Secondary	Tertiary	Slimes
LOM production	Mm^3	29.08	6.48	18.92	64.89
Specific gravity of solids	t/m^3	3.03	3.03	3.03	3.10
Process moisture content	%	13	15	15	32
Geotechnical moisture content	%	15	18	18	47
Solids content	%	87	85	85	68

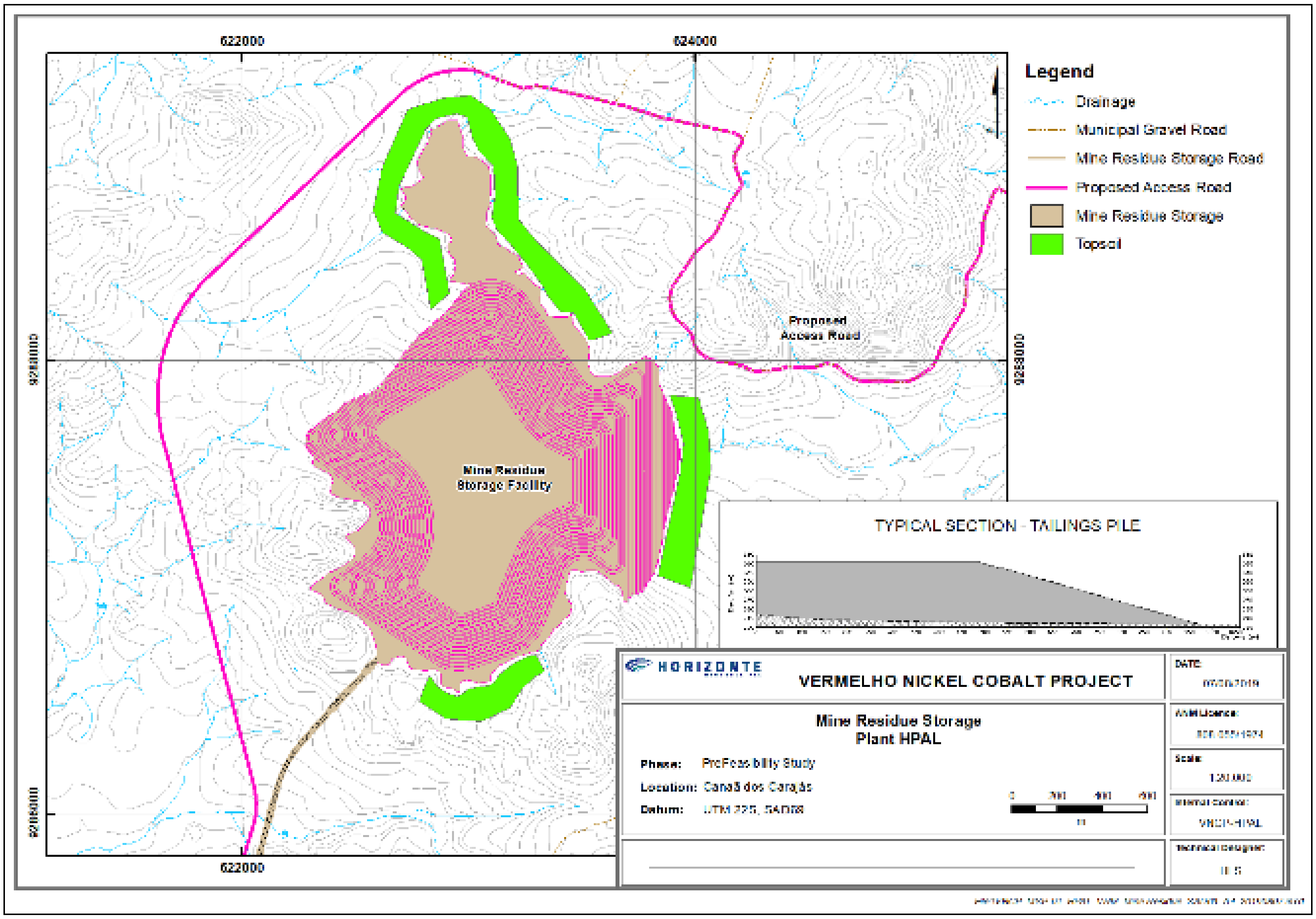
The facility will be constructed using the "Ascending" approach wherein the primary and secondary products are compacted and used to construct a "ring" dyke within which the tertiary and slimes are placed. This approach was adopted because the primary and secondary are compactable and well-draining, whereas the tertiary and slimes are less so and should be retained.

WALM decided to adopt a more conservative arrangement, with a flatter overall slope angle of about 3.5H:1.0V, since the material to be dry stacked is essentially composed of fines (tertiary tailings and slimes), highly erodible and susceptible to undrained loading. The general layout of the facility is shown in Figure 18-5.

A key feature of the facility is adequate draining on top of and under the stack. To facilitate this the facility will be lined with a primary liner layer of clay soil compacted to a minimum thickness of 0.50 m, located in the lower portions and on top of the recharge underdrain, or, in the steeper portions of facility the primary liner will be bentonite geo-composite. On top of this a secondary liner will be placed. The secondary liner is a 1.5 mm thick double textured high-density polyethylene (HDPE) geotextile.

WALM designed the drainage system to collect the stormwater and convey it to the downstream sediment retention dyke, thus preventing the development of erosive processes and the transportation of fines to the downstream watercourse.

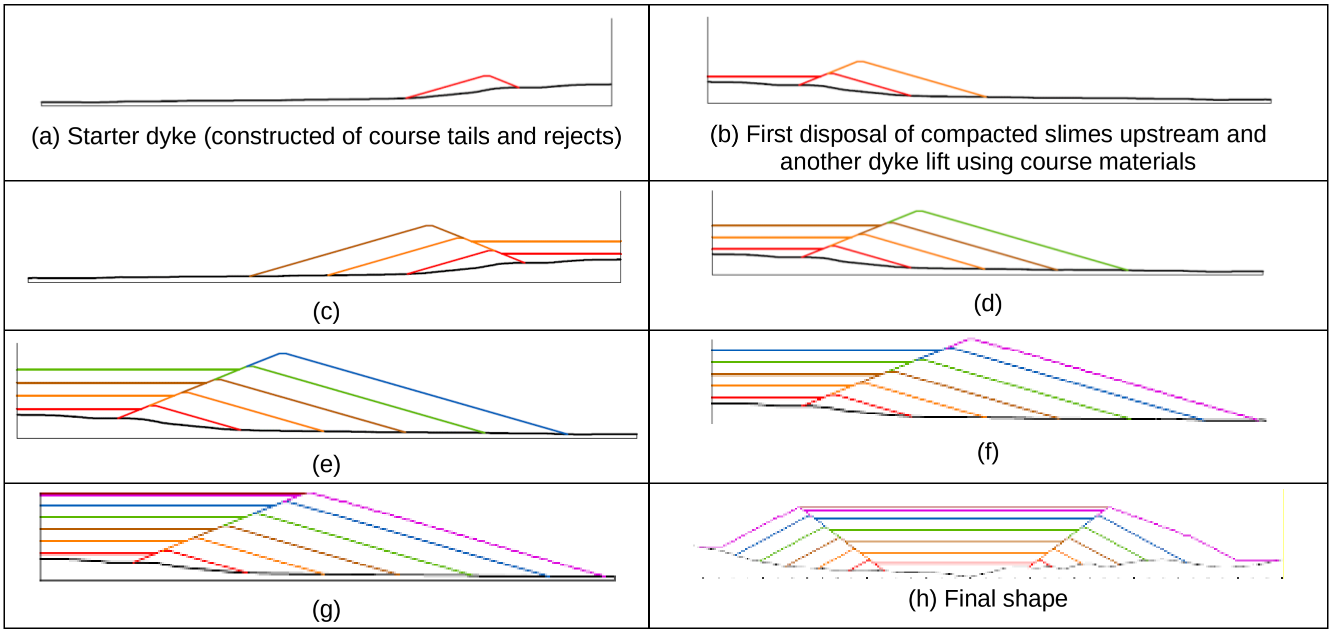
Figure 18-5 Mine residue storage facility



Source: HZM, 2019

After the liners and drainage system is in place, the mine residue will be placed in the sequence shown in Figure 18-6 below.

Figure 18-6 Deposition sequence of mine residue facility



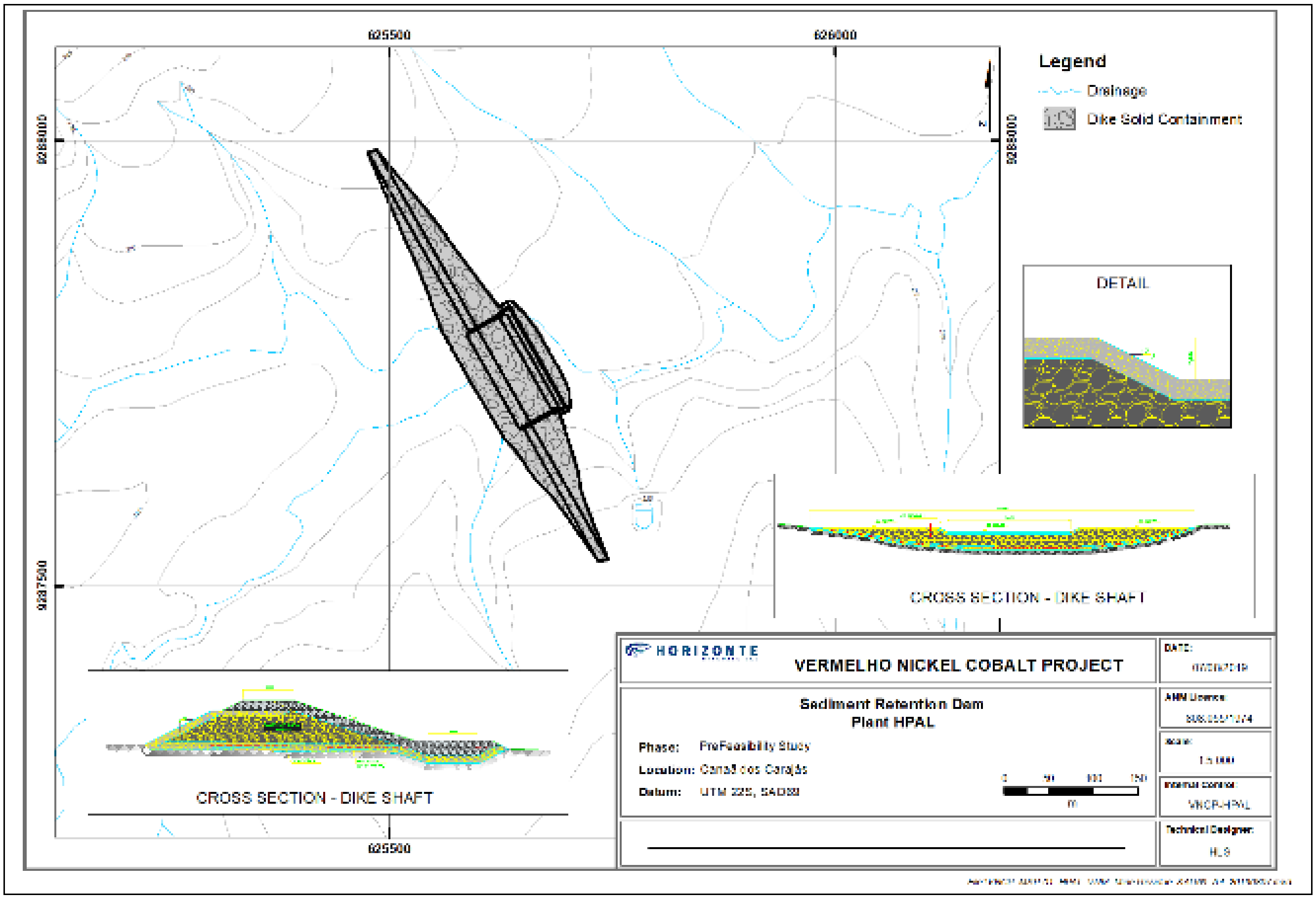
Source: WALM, 2019

WALM estimated initial costs and ongoing costs for the construction and maintenance of the facility. These costs are detailed in Section 21.

18.5 Sediment retention dam

For retaining sediments generated from the mine residue storage facility, an over-toppable rock filled filter dam will be constructed (Figure 18-7).

Figure 18-7 Sediment retention dam



Source: WALM, 2019

The key design features of this dam are as follows:

- Upstream slope: 2.0 (H): 1.0 (V)
- Downstream slope: 30 (H):1 (V) across the embankment and overflow channel.

Although the body of the dam has been designed to perform as a drain, a spillway system will be installed to properly convey runoff to the environment thus ensuring structure's integrity. This spillway system consists of the following elements:

- A trapezoidal overflow weir with a minor base of 120.0 m, 2.00 m in height, side slopes of 2.0 (H):1 (V), stone pitched facing, mean block size (D_{50}) = 50 cm
- A pilot channel, corresponding to the body of the dyke, stone pitched facing, mean block size (D_{50}) = 50 cm, longitudinal grade of 3.0 (H):1 (V) and cross slope of 2.0 (H):1 (V)
- A settling basin to reduce the erosive potential of the discharged flows, stone pitched facing, mean block size (D_{50}) = 50 cm.

WALM completed a cost estimate for the dam. These costs are detailed in Section 21.

18.6 Energy supply for the Project

The electrical demand for sizing of the substation and the transmission line was established from the estimated electrical load; a summary is shown in Table 18-6.

Table 18-6 Estimated electrical load – installed capacity

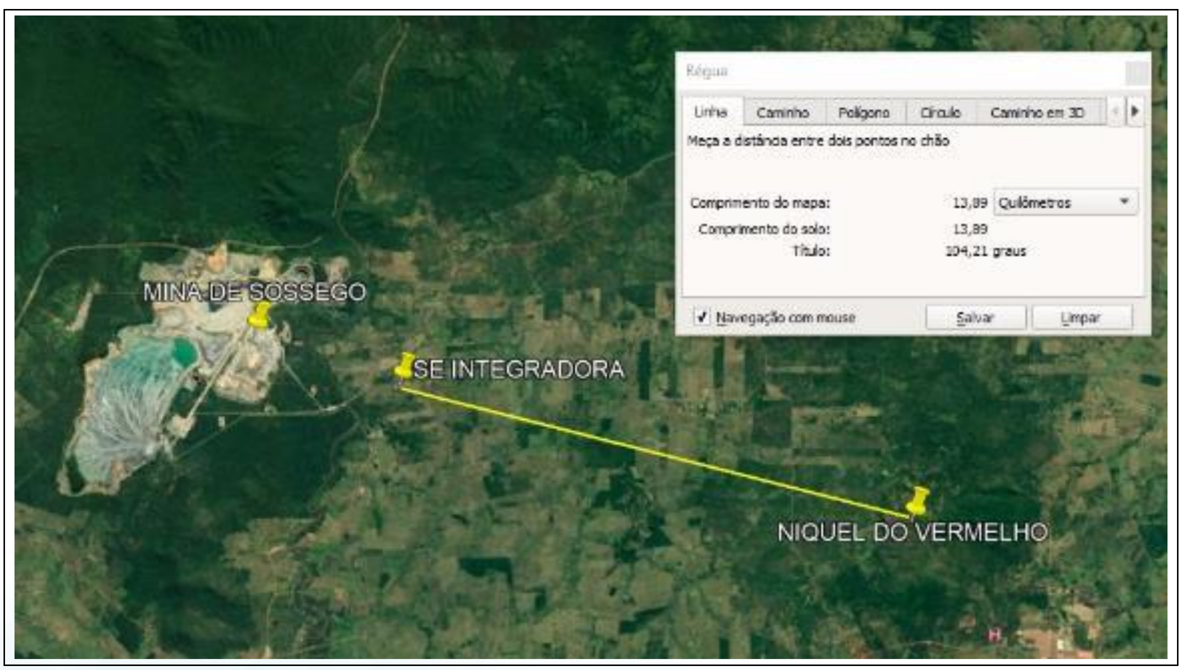
WBS level 1	WBS description	Installed power (kW)
100	Feed preparation	5,348
200	Leach	14,548
400	Nickel and cobalt recovery	4,300
500	Wastewater	2,887
600	Reagents	29,589
700	Water	4,031
800	Utilities	1,327
Total		**62,022**

Based on the installed capacity and estimated mechanical utilisation, Simulus determined that the peak power draw for the plant will be 37,988 kW. The plant includes a co-generation system attached to the acid plant; this generates 23,922 kW when in operation, making the plant net power draw 15,056 kW.

SM&A completed a study for the supply of power to the Project. SM&A concluded that there is sufficient supply of hydroelectric power in the region to meet the demands of the Project along with considerable local infrastructure to support it.

SM&A reviewed existing electrical substations close to the Project (Figure 18-8) and it was concluded that the nearest substation is SE INTEGRADORA (about 15 km). This 230-kV substation belongs to Eletrobrás/Eletronorte and is supplied by lines from Carajás, as well as feeds and/or connected to other substations such as the Sossego Mining, Onça Puma Mining, the S11D Project and the Xingura-2 substation which, coincidentally, it is the energy source for the HZM's Araguaia project.

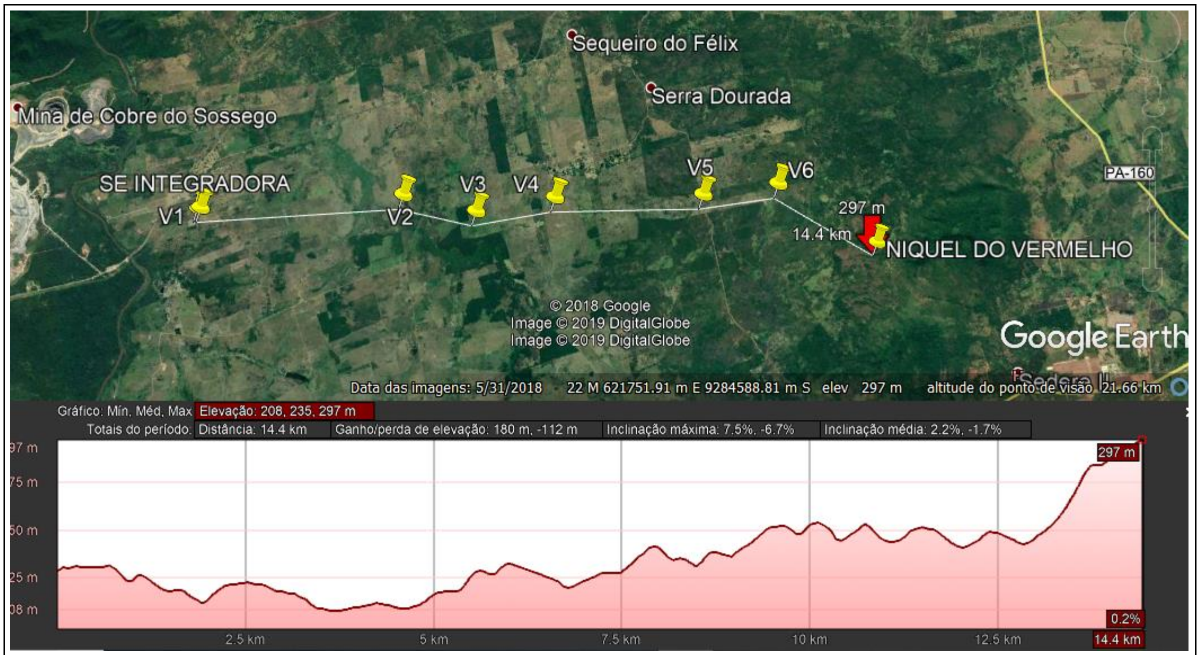
Figure 18-8 Electrical substations near the Project



Source: SM&A, 2019

Consequently, SM&A determined that energy for the Project could be delivered by connecting to the SE-Integradora substation owned by Eletrobrás/Eletronorte. The solution includes construction of a new 230 kV bay at SE-Integradora, a new 15 km 230 kV transmission line (Figure 18-9) and a new substation in the Project, with one entrance bay and two or three transformers bays.

Figure 18-9 Proposed transmission line route



Source: SM&A, 2019

SM&A completed a capital cost estimate for the powerline and the substations; this is detailed in Section 21.

19 MARKET STUDIES AND CONTRACTS

In June 2019, HZM commissioned Wood Mackenzie (WM) to develop a report on the market for nickel sulphate. It is summarised along with the cobalt sulphate market in Sections 19.1 and 19.2 below.

19.1 Nickel

19.1.1 Introduction

Nickel belongs to the transition metals and is hard and ductile. Pure nickel, powdered to maximise the reactive surface area, shows a significant chemical activity, but larger pieces are slow to react with air, under standard conditions because an oxide layer forms on the surface and prevents further corrosion. The metal is valuable chiefly in alloys, and in decreasing magnitude of production: stainless steel production, then nickel-based and copper-based alloys, steel alloys, foundries, plating and the battery sector. As a compound, nickel has several niche chemical manufacturing uses, such as a catalyst for hydrogenation, cathodes for batteries, pigments and metal surface treatments.

19.1.2 Sources of nickel supply

According to WM, global mined nickel production in 2018 was 2.2 Mt (of contained nickel). About 60% originates from sulphide ores and 40% from laterites. Sulphide ores account for about 30% of global resources and as the larger nickel sulphide deposits deplete, the proportion of ferronickel as a source of nickel units in the market is forecast to increase, because most of the new projects coming on stream are predominantly nickel laterite, processed through either hydro- or pyrometallurgical routes, the latter primarily producing ferronickel.

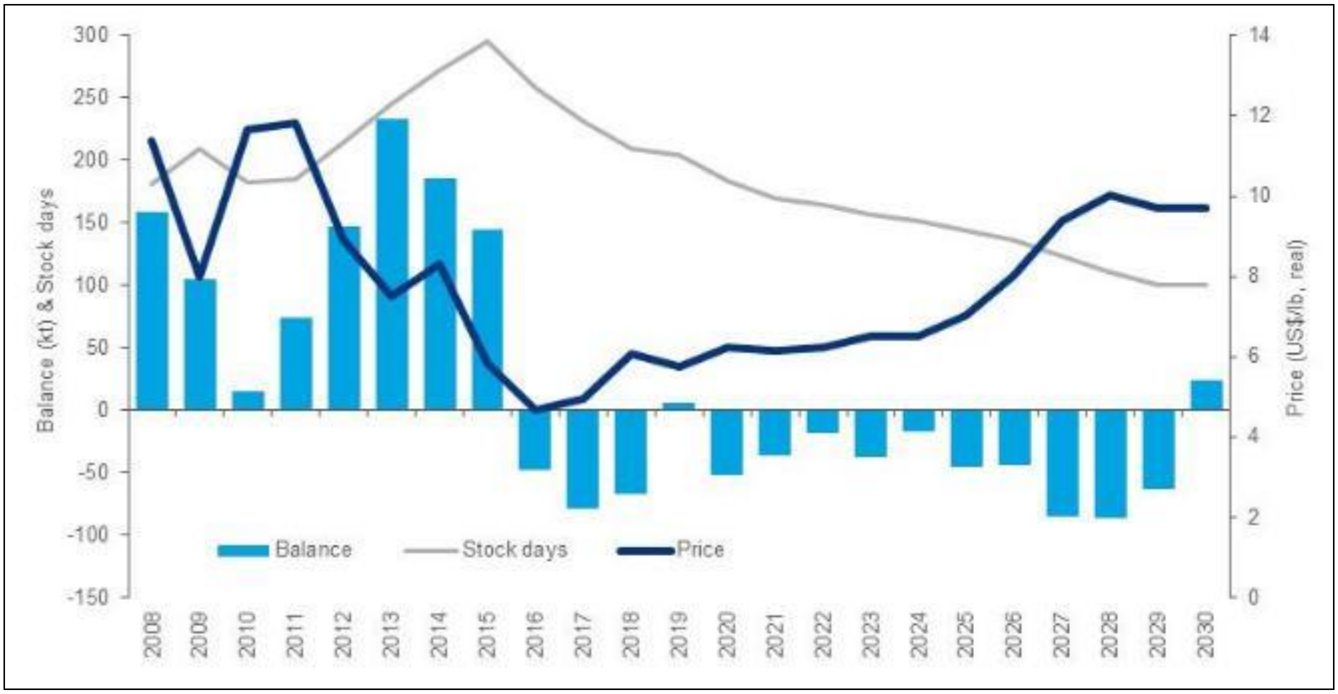
19.1.3 Outlook for nickel

After many years of annual surpluses, the global nickel market moved into deficit in 2016. Although the shortfall in that year was moderate, it was followed by larger deficits in 2017 and 2018. WM believes that the ongoing ramp-up of nickel pig iron production in both Indonesia and China will leave the nickel market with a small surplus at the end of 2019, before further consecutive deficits resume from 2020. As the shortfall accumulates, so the large inventory excess that had built up over the prior years of oversupply will be drawn down. This is most visibly represented by the ongoing decline in exchange nickel stocks on the LME and Shanghai Futures Exchange (SHFE).

World nickel demand increased by 4.6% in 2018, to 2.28 Mt, and is expected to increase by a similar amount in 2019, to 2.39 Mt. After that, growth at a compound annual growth rate (CAGR) of 3% will raise world demand to 2.84 Mt in 2025, supported by further expansion in Chinese stainless-steel capacity between now and 2022. Thereafter, growth in demand will slow to a CAGR of approximately 2.2%, to 3.16 Mt in 2030, before accelerating uptake by the battery segment boosts the CAGR to 2.6% through to 2035. By then, world nickel demand will stand at 3.59 Mt. Over this period, primary nickel uptake in stainless will account for a declining portion of total demand, moving from 70% to 50%, yet rising in volume terms from 1.64 Mt in 2019 to 1.86 Mt in 2025, and 1.93 Mt in 2035 (Figure 19-1).

China will remain the main driver of demand over the long-term, by virtue of its stainless-steel industry, supported by the development of a new stainless-steel industry in Indonesia, where Tsingshan now has 3 Mt/a of available melt shop capacity. Asian demand for nickel will receive a further boost from the expansion in demand for nickel sulphate ($NiSO_4$) in battery cathode materials, chiefly for electric vehicle (EV) batteries. Currently, Wood Mackenzie estimates that Asian companies account for approximately 80% of global $NiSO_4$ production.

Figure 19-1 Nickel balance, price and stocks



Source: Wood Mackenzie, 2019

19.1.4 Nickel sulphate market

Nickel sulphate is a hydrous salt commonly sold as green or blue crystalline solid. It has the chemical formula $NiSO_4 \bullet 6H_2O$ and contains approximately 22.3% Ni by weight. It is most commonly used in solution as the electrolyte in nickel plating or as one of the key feed chemicals used in the production of battery cathode materials, most notably in lithium-ion batteries for hybrid and EVs. Nickel sulphate has other minor chemical applications, such as in the making of catalysts.

Nickel sulphate is one of a number of nickel chemicals that are produced by conventional refining. Thus, Nornickel (at Harjavalta) and Sumitomo Metal Mining (at Niihama and Harima) both produce nickel sulphate in lesser quantities to nickel metal.

Aside from conventional refining, the process of making nickel sulphate is essentially one of using heat to dissolve nickel in sulphuric acid. Plating and battery cathode manufacture requires nickel sulphate of very high purity; the most logical way to go about this is to use pure nickel metal, including Class 1 cathode, cut cathode, briquettes, pellets and powder. The dissolution of each form has its own energy and time requirements. In general, solid cathode needs more heat and takes more time to dissolve than briquettes and pellets and is therefore more costly. Nickel powder reacts the fastest but is undermined by the rapid generation of hydrogen gas. Thus, pellets and briquettes are considered the optimum forms used in the industrial conversion of nickel metal to nickel sulphate.

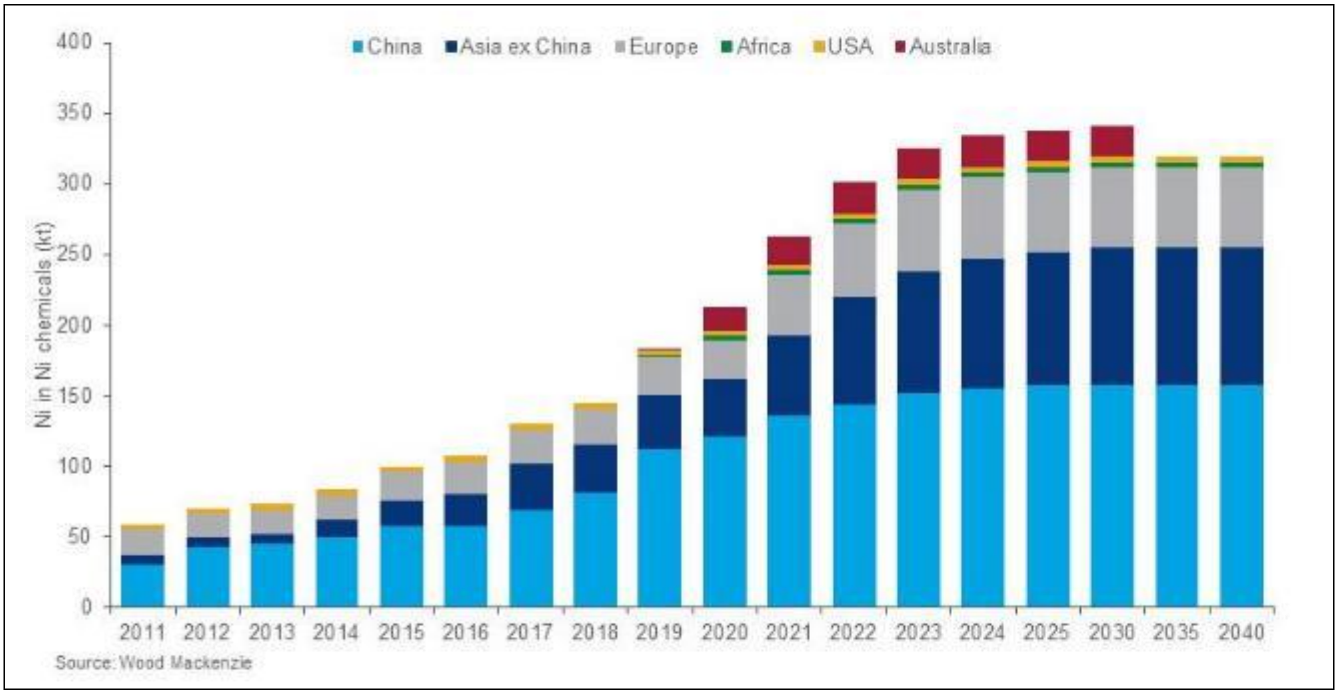
It is also possible to react other nickel-containing materials with sulphuric acid to produce nickel sulphate, although, in this case, a purification step is needed to produce a clean nickel sulphate of acceptable specification.

19.1.5 Nickel sulphate supply

World nickel sulphate production has expanded considerably in recent years (Figure 19-2). Prior to 2012, global nickel-in-chemicals production was steady at 30–50 kt/a. However, since then, growth in supply has advanced at a CAGR of 13%, to 144 kt in 2018, largely due to expansions in nickel sulphate capacity in China, Japan and Taiwan. The driving force behind these investments in new supply is the growing interest in producing batteries for EVs. Thus, while the wider nickel market has only become focused on the advancement of nickel use in EV batteries over the past 18 months to two years; Asian countries have undertaken testing and research since 2012.

Most of the 144 kt of nickel in chemicals production in 2018 was as nickel sulphate. Approximately 83% of this total was primary output produced by conventional refining or processing utilising a combination of raw materials, including mixed intermediates, scrap and crude nickel sulphate. The remaining 17% is from "converters" that dissolve nickel metal in sulphuric acid.

Figure 19-2 $NiSO_4$ production by region



Source: Wood Mackenzie, 2019

Regionally, more than 50% of production is based in China, with a further 22% in South-East Asia, principally Japan and Taiwan. Most of the remaining production is located in Europe.

Production is expected to increase by 28% in 2019, to 184 kt, mostly due to further expansion in China and the ramp-up of new capacity in South Korea (Kemco), Australia (BHP's Kwinana sulphate project) and South Africa (Thakadu). A second non-Asian facility, Terrafame (Finland), is also expected to commence the production of nickel and cobalt sulphate from its current mixed sulphide product, starting in 2021.

19.1.6 Nickel sulphate demand and substitution

Nickel sulphate is mostly used in two applications: as a chemical feed for the production of battery cathode materials; and as the electrolyte in nickel plating. Comparatively minor quantities are also used in other chemical applications, such as the production of nickel-containing catalysts (e.g. nickel-cobalt (NiCo) catalysts used in petroleum refining).

The division of production that goes to the plating or battery segments varies from country to country. In China and South-East Asia, producers supply consumers in both industries, whereas in the US, the two producers have principally supplied only the plating sector because of the late development of battery chemicals manufacturers. Historically, platers received an estimated 60% of Chinese production, however with the recent expansion in production to address the battery sector demand the share of consumption from plating has decreased and is expected to decrease further as EV battery demand increases.

It is possible that alternative nickel chemicals to nickel sulphate could be used in plating and battery applications, but it is the ease of production that is important. In plating, the existing share of the market held by alternatives is very small, such that further penetration is likely to have a negligible impact. For battery production streams, any substitution by other chemicals is likely to be determined by changing battery chemistry requirements and cheaper production routes for such alternative chemicals.

Plating

A wide range of materials can be plated with nickel, either to impart corrosion resistance or to enhance visual appeal, as for jewellery. The demand for nickel-plated products is mainly driven by decorative appeal. Historically, most articles that required plating were metallic, particularly carbon steel, but today coating plastic rather than metal has become the industry norm.

Nickel plating is used in a variety of applications, such as building products, auto/bike components, electronics, jewellery, fittings, haberdashery (pins and studs), engineering repairs, electroforming, printing plates and stampers (e.g. for DVDs and CDs). Principal end-use segments are automotive and household appliances, fixtures and fittings, the latter being primarily determined by fashion. In autos, plated plastic components include badges and logos, interior parts and some exterior trim, including high performance wheels.

Nickel is used in two principal forms in the plating industry, as a metal and as nickel sulphate. Small amounts of other nickel chemicals are also used, such as nickel chloride ($NiCl_2$), nickel carbonate ($NiCO_3$), and nickel sulphamate ($Ni(SO_3NH_2)_2$). In 2018, approximately 78% of the nickel units for plating were sourced from nickel metal, 20% from sulphate and the balance from nickel chloride and sulphamate.

In electroplating, nickel metal is used as the anode, ideally as 1x1” cut cathode or pellets, contained in a titanium wire basket. At the same time, nickel sulphate is used as the electrolyte, facilitating the transfer of nickel ions from the anode to the article to be plated, on the cathode. An alternative process, electroless plating, does not involve the passing of an electric current between electrodes. Instead, the article to be coated is dropped directly into a bath of hot nickel sulphate mixed with sodium hypophosphite. This triggers an autocatalytic reaction that produces a uniform, highly even deposit of a nickel-phosphorus alloy on the article.

WM estimates that nickel demand in plating was approximately 140 kt in 2018, or about 6% of world nickel demand. Nickel demand in plating applications is projected to increase steadily over the mid to long-term, at a CAGR of approximately 2%, with demand estimated at 158 kt in 2025 and 187 kt in 2035.

China consumes the most plating nickel, accounting for 54% of total volumes, with a further 25% consumed by other South-East Asian countries.

Batteries for electric vehicles and energy storage

Nickel sulphate is an important feed chemical in the production of nickel batteries, such as nickel metal hydride (NiMH), nickel cadmium (NiCd) and some variants of lithium ion (Li-ion), including NCM (Ni-Co-Mn) and NCA (Ni-Co-Al). NiMH batteries have long been the preferred power train in hybrid vehicles, but it is the anticipated future dominance of full battery electric vehicles (BEVs) that is driving the expansion in nickel sulphate production.

The current consensus opinion is that over the medium-term, NCM and NCA Li-ion type battery chemistries will be favoured for BEVs. This does not mean that all BEVs will be powered by batteries containing nickel, as other types, such as lithium iron (Fe) phosphate (LFP), which is widely used in China, will remain in use. The projections for BEV fleet expansion and battery use indicate that the quantity of nickel which will be required in the long-term will be substantial. This will be supplemented by the expected longer term need for energy storage batteries, which are likely to depend on the same battery types.

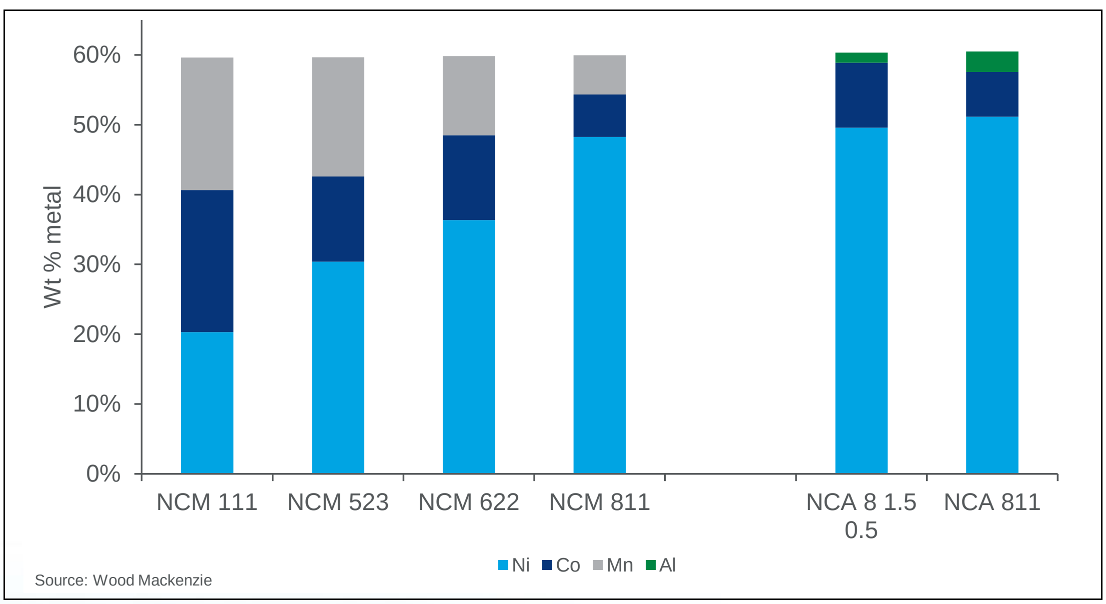
WM's outlook assumes that all nickel used in EV and energy storage batteries starts as nickel sulphate, as this is the key chemical raw material needed to make the precursor compounds from which battery cathodes are made. Medium-term projections are partly guided by the ramp up of nickel sulphate production, as WM consider that the expansion in output since 2012 is primarily a result of the anticipated increase in demand for EV batteries.

Over the longer term, WM use a top-down approach that is founded on WM projections of EV sales, including hybrid electric vehicles (HEVs), plug-in hybrid electric vehicles (PHEVs) and full electric (BEVs). The calculation of the quantity of nickel consumed in the batteries depends on a large number of factors, each requiring their own projection, the most critical ones being:

- Size of the battery needed
- Types of battery used across the fleet
- Nickel content of the battery.

Li-ion NCM batteries may be the future battery of choice for most car makers but evaluating their future nickel content is made more complicated by the different formulations within that category alone (Figure 19-3). These are determined by the ratio of nickel to manganese to cobalt in the lithium oxide base of the cathode material. The most common cells in use currently, are type 111 (20% Ni by weight in the cathode) and type 532 (30% Ni). Technical advances will likely mean increasing penetration of type 622 (36% Ni) and ultimately type 811 (50% Ni). However, as increasing the nickel component in these formulations also lowers the threshold of thermal runaway (i.e. increases the potential for spontaneous combustion and therefore the risk of fire), it remains highly contentious as to when the widespread adoption of NCM811 batteries for cars will be possible.

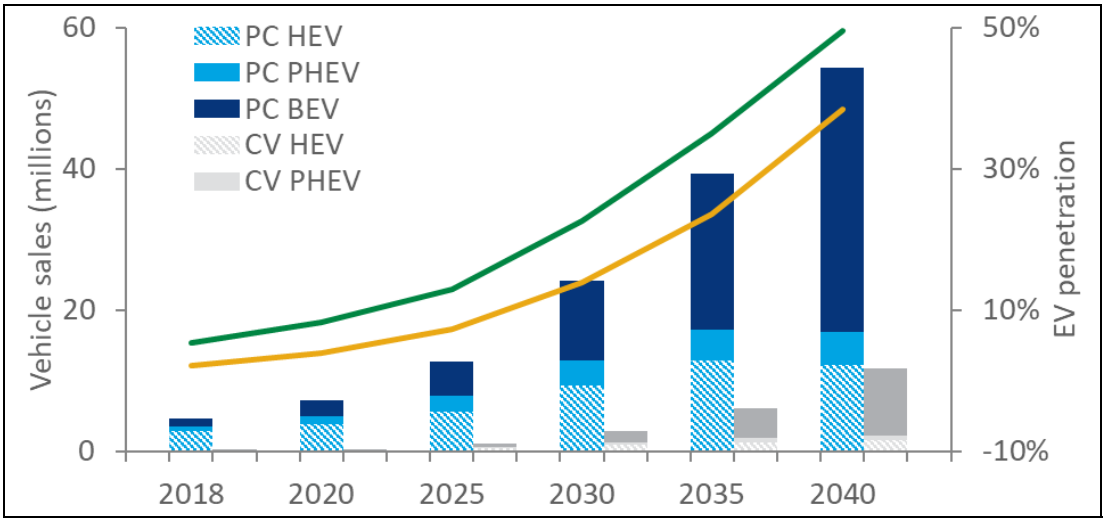
Figure 19-3 Metal content of ternary battery cathodes



Source: Wood Mackenzie, 2019

WM forecasts that the world fleet of passenger EVs will increase from 4 million in 2017 to nearly 60 million in 2040 (Figure 19-4). This represents an increase in EV share of the total passenger car fleet from approximately 5% to 50% over the period. Penetration of BEVs is significant after 2025, with their share of all passenger cars increasing from approximately 6% in 2025 to 35% in 2040. The number of commercial vehicles using electric batteries will also start to accelerate after 2025, but the size of this fleet remains small in comparison with passenger cars.

Figure 19-4 Vehicle sales forecast



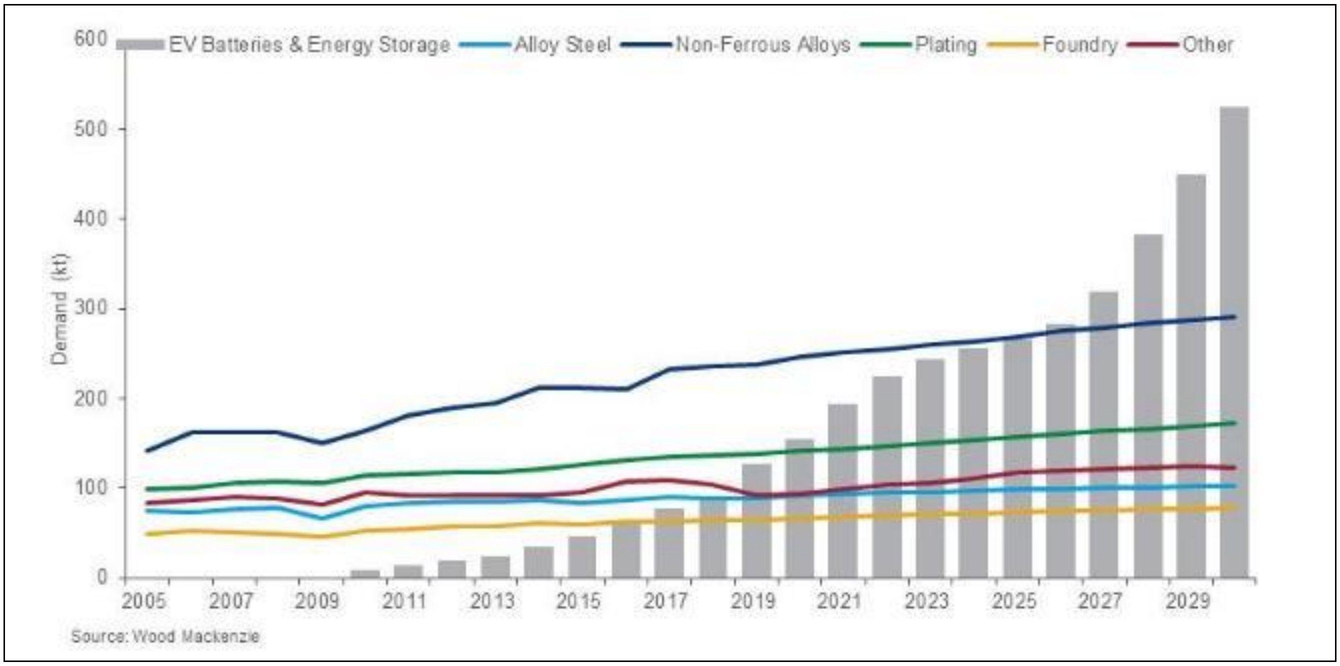
Source: Wood Mackenzie, 2019

WM estimates that NCM (35%) and NCA (20%) batteries were used in about half of all EVs in 2018. The balance was taken by LFP (30%) and other types (15%). WM expects that LFP will retain a strong position in the market over the long-term, particularly in commercial and low-end vehicles. The anticipation is that NMC will increase its share of the market to approximately 70% in 2040, with NCA steady at approximately 20% and LFP at 10%.

Given the aforementioned safety issues surrounding NCM811, WM consider that mass adoption will be in the long-term, that uptake will be gradual, and vary depending on the level of electrification. Outside China, there are no models currently using NCM811. Thus, NCM111 and NCM532 formulations accounted for approximately 90% of all NCM batteries in EV use in 2018. Through to 2025, NCM622 will become increasingly dominant (to over 50% of the market in 2025). By 2025, WM expects NCM811 to be widely used, but it is not until 2030 that this nickel-rich cathode type takes over from NCM622.

Figure 19-5 illustrates WM's outlook for non-ferrous nickel consumption.

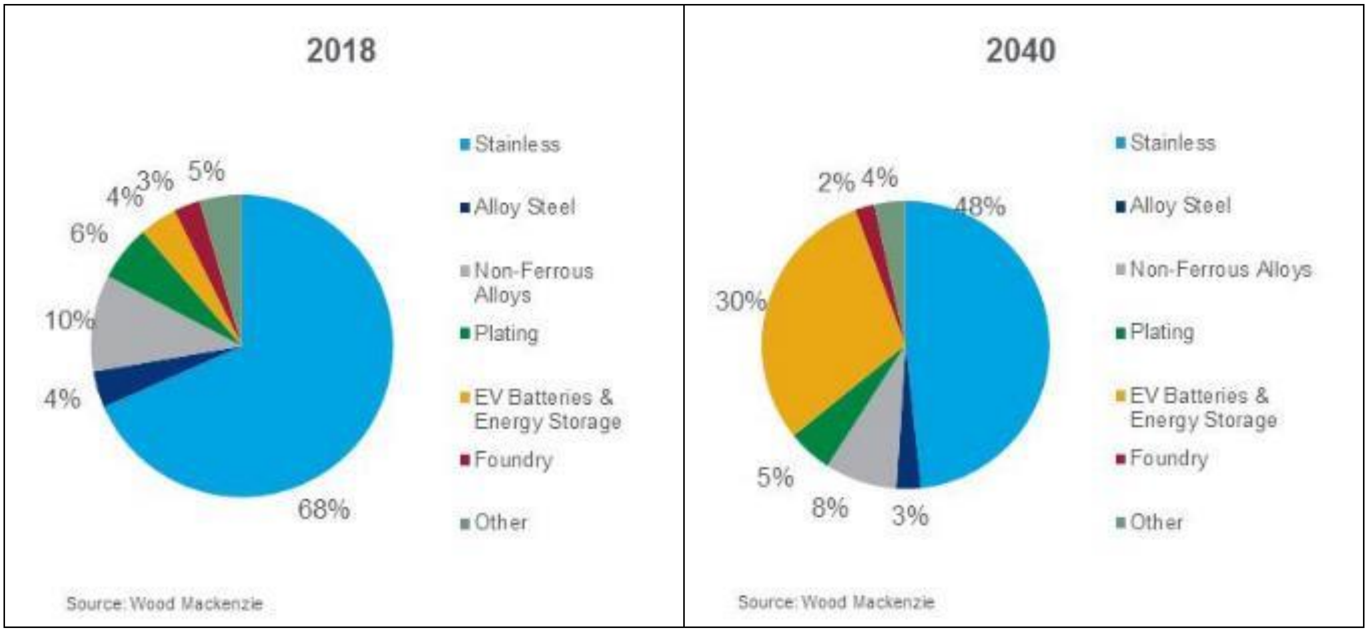
Figure 19-5 Nickel consumption in non-stainless first uses



Source: Wood Mackenzie, 2019

Figure 19-6 illustrates WM's outlook for nickel demand trends in non-stainless first uses.

Figure 19-6 Share of demand (2018 and 2040) shifting towards EVs



Source: Wood Mackenzie, 2019

19.1.7 Nickel sulphate prices and premia

Traditionally, premiums on nickel sulphate have been largely determined by customers using it as an electrolyte in the plating industry. High purity nickel sulphate receives a premium price in the marketplace and is essential for electroplating. Other determinants include whether the sulphate is received in crystal or liquid form.

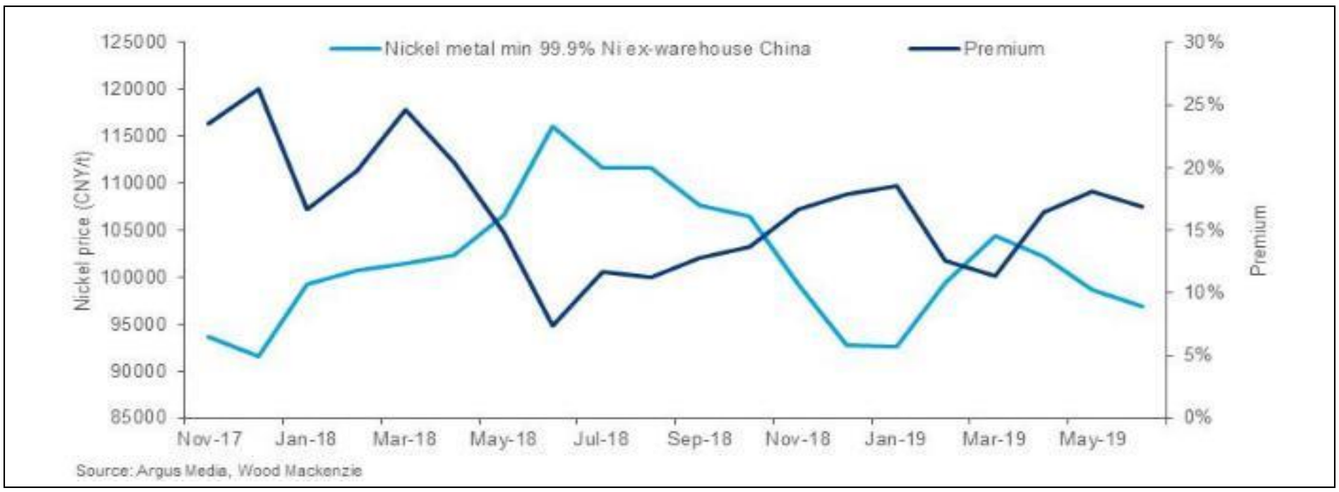
Plating is a highly specialised market, with few direct sales being made by the manufacturer to the consumer because of the disparate nature of the sector and the small size of many consumers. Thus, sales are handled mostly by traders and agents who set the price according to their client's specific requirements, which may include the need for a specific nickel product or brand, or the need for a generic nickel sulphate for blending into a higher value products (e.g. in electroless plating).

As plating is a niche and disparate sector, it is reasonable to expect that selling nickel sulphate in this market will continue to achieve premia, although it may be the intermediary seller that benefits rather than the producer. Premiums are highly variable and difficult to compare in the nickel-sulphate-in-plating segment.

Until recently, nickel sulphate was important only in the niche plating segment (which accounts for a relatively small portion of world nickel demand); pricing for nickel sulphate had not received much attention globally (outside of plating sector). The rapid rise in demand for EVs and associated batteries has allowed for greater prominence of nickel sulphate pricing.

Visibility on nickel sulphate pricing is limited to China. Since November 2017, Chinese calculated premiums on nickel sulphate (minimum 22% Ni; ex-works) have ranged between 6% and 30% of the Chinese class 1 nickel price (ex-warehouse). Through the second half of 2018, premiums increased, peaking at around 20% at the start of 2019. Although falling back to around 10% in March 2019, premiums have since rallied to approximately 17% to 18% in May and June 2019 (Figure 19-7).

Figure 19-7 Chinese $NiSO_4$ premiums



Source: Wood Mackenzie, 2019

The strength of these premia is consistent with the stronger demand for nickel sulphate from battery precursor manufacturers. At the same time, they also reflect higher premiums on other nickel products, such as powder, pellets and briquettes, which have been sought in increasing quantities to dissolve as raw materials for nickel sulphate production. However, it is also important to note that the increase in premia is partly related to the decline in nickel price from mid-year 2018.

WM does not forecast premiums. There is no guarantee that current high premiums may equate to similarly high premiums in the foreseeable future. As the demand for nickel sulphate from the battery segment is expected to accelerate, it is feasible that premiums will be realised in the medium- and long-term. However, WM notes that a material increase in nickel sulphate capacity and supply in China and South-East Asia, and at Kwinana (Australia) and Terrafame (Finland), may affect pricing (and premiums) negatively.

More recently, a group of HPAL projects in Indonesia intend to process MHP directly to a nickel sulphate. Most of this will be sold to partner companies in China for making battery precursors, whose purchasing costs should be lower than buying sulphate from sources using traditional processing methods. These projects raise the potential for many more such plants being built over the medium-term. Therefore, one risk to future premiums is that the recent trend for Class 1 nickel (for dissolution into nickel sulphate) may dissipate as the supply of battery feed chemicals from these integrated HPAL (or similar hydrometallurgical) facilities gathers momentum. Both the integrated nature of these plants, the greater abundance of nickel sulphate and reduced demand for Class 1 nickel, would lower market support for high nickel sulphate premiums.

19.1.8 Economic evaluation price and sulphate premium

HZM considers a reasonable near-term class 1 nickel price forecast for the purposes for the PFS is the CIBC consensus price of, US$16,400/t (US$7.44/lb) – this was applied in the Base Case model along with a US$2,000/t (US$0.91/lb) premia for production of nickel sulphate (which is of higher value in the battery market than class 1 nickel).

The WM long-term incentive price currently is approximately US$19,800t (US$8.98/lb). In selecting the consensus nickel price of US$16,400/t, a conservative approach was adopted. A fixed price for nickel was applied over the LOM. The Qualified Person has reviewed the above and considers that the results support the assumptions in this Technical Report.

19.2 Cobalt

19.2.1 Cobalt supply and demand forecast

Cobalt is used in Li-ion battery precursor production as a sulphate or as an oxide. In addition, cobalt metal powder and cobalt hydroxide are used to produce rechargeable alkaline batteries.

Cobalt demand for use in batteries is forecast to grow at an annual rate of 15.6% from 52,500 t in 2016 to 215,000 t in 2025. Total global cobalt demand is forecast to increase from 99,000 t in 2015 to 274,000 t in 2025.

Cobalt supply is constrained by several factors. It is typically produced as a by-product at nickel and copper mining operations, so is inextricably linked to demand for nickel and copper and supply is less sensitive to cobalt prices. Over 60% of global cobalt production is sourced from the Democratic Republic of Congo (DRC). Media and non-government organisation (NGO) reports have documented claims of human rights abuses and child labour at artisanal cobalt mines in the DRC. Calls for increasing supply chain transparency from NGOs is pushing major end users of Li-ion batteries including Apple Inc., Tesla Inc., General Motors Company and Volkswagen AG, to focus on the audit trail of the cobalt supply chain.

There is widely expected to be a supply/demand gap as use in battery technology increases over the next 10 years. Copper mines in the DRC and Zambia contribute 74% of the mined production of cobalt, to meet this demand/supply gap will require more than doubling output from the African Copper Belt. While this is not unlikely (given that the copper market is expected to be in short supply by 2025), it is more likely that a prolonged cobalt shortage will lead producers to experiment with lower cobalt content chemistries in rechargeable alkaline battery manufacture. It is reported that lower cobalt content in these batteries may eventually bring the market back into balance.

The Project is expected to be a globally significant producer of cobalt.

19.2.2 Cobalt and cobalt sulphate price

In the study a flat LME/London Metal Bulletin cobalt price of US$34,000/t (US$15.4/lb) for the LOM has been assumed. This assumption of US$34,000/t is significantly below the long-term consensus forecasts of US$55,000/t (US$25/lb).

Deviations both above and below the LME price occur because of short-term demand and supply imbalances. For the PFS, it has been assumed that the cobalt sulphate price (metal equivalent basis) will be equal to the LME price of cobalt.

19.3 Kieserite

In July 2019, HZM commissioned a report on the market for kieserite in Brazil from Dr Fabio Vale (Diretor Técnico/Technical Manager) of Adubai Consultoria Agronômica (Adubai). The report is summarised below.

19.3.1 Background

Kieserite is a naturally occurring mineral that is chemically known as magnesium sulphate monohydrate. The chemical properties of kieserite are:

- Chemical formula: $MgSO_4•H_2O$
- Mg content: 16% (kieserite fine); 15% (kieserite granular)
- S content: 22% (kieserite fine); 20% (kieserite granular)
- Solubility: 417 g/L (20°C)
- Solution pH: 9.

Kieserite provides a highly concentrated form of two essential plant nutrients – Mg and S. Since kieserite applications have no major effect on soil pH, it can be supplied to all kinds of soil, irrespective of soil pH. It is commonly used prior to or during the growing season. Due to its high solubility, it can be used to supply both Mg and S during peak periods of crop demand.

Kieserite itself is not used as foliar fertiliser or in fertigation systems, but it serves as raw material to produce Epsom salt ($MgSO_4•7H_2O$), which is completely soluble, and suitable for both fertigation and foliar application.

Sulphur in plants helps form important enzymes and assists in the formation of plant proteins. Sulphur deficiencies can cause major plant health problems and loss of vitality.

Magnesium is very important in plant photosynthesis. Without magnesium, chlorophyll cannot capture the energy from the sun needed for photosynthesis. Magnesium is also used by plants for the metabolism of carbohydrates and in the cell membrane stabilisation.

19.3.2 Current production

Kieserite is primarily obtained from deep underground deposits in Germany. The fine crystalline kieserite is sold for direct application to soil, or it is granulated to a larger particle size (for mechanical fertiliser spreading or for bulk blending with other fertilisers).

Currently, the high production costs have limited the market for kieserite in Brazil and most production is consumed in Europe (agriculture).

19.3.3 Potential uses for Kieserite in Brazil

Owing to very long periods of warm rainfall Brazilian soils in general are extremely poor in macronutrients, mainly in Cerrado[9] areas, and the use of fertilisers is very important for growers to achieve economic yields.

Kieserite is of particular benefit to crops that have an agronomic response to sulphur and magnesium. These would likely be palm oil, coffee, vegetables, cotton and soybean. Currently, palm oil and coffee are the largest fertiliser consumers in Brazil. The company, Adubai, developed an agronomic trial to evaluate kieserite as fertiliser for the coffee crop in São Paulo state between 2013 and 2016. In this trial, it was demonstrated that application rates between 150 kg/ha to 300 kg/ha of kieserite were most effective.

For cotton and soybean, despite the high possibility of a positive response to the two nutrients, there has not been a trial to demonstrate this in Brazil. Carvalho *et al.* (2009) concluded that kieserite had a positive effect on the cotton crop, improving the nutrition in Mg and S, increasing the boll retention and cotton yield. The rate between 240 kg/ha and 280 kg/ha of kieserite presented the best results.

19.3.4 Fertiliser market in Brazil and Kieserite

The fertiliser market in Brazil is large. In 2018, 35.6 Mt of fertiliser was sold; of this, 77.5% was imported and 22.5% was manufactured locally. The amount of kieserite sold in Brazil is very low in comparison (mainly due to its high cost). The quantity does not appear in official fertiliser import statistics. The author estimates that between 15,000 t and 20,000 t of kieserite was imported in 2018. From this total, it is estimated that 75% is consumed in Pará state and the rest is distributed to other states, specifically to São Paulo and Minas Gerais for coffee, vegetables and other special fruits. The expectation in 2019 is for similar volumes to be sold.

The Pará state market for kieserite is for palm oil plantations. Palm oil has a very high demand for both magnesium and sulphur. It is common for farmers to apply magnesium at a rate of approximately 10–15 kg/ha, and sulphur at approximately 50–60 kg/ha. For example, one common blend used in the palm oil is industry is "12-09-21 + 1,2% Mg + 6,5% S", which comprises 190 kg/t of ammonium sulphate, 185 kg/t of ammonium nitrate, 175 kg/t of monoammonium phosphate, 350 kg/t of KCl, and 100 kg/t of kieserite. The commercial palm oil crop in Pará state is approximately 150,000 ha. Given the application rate of kieserite is 100 kg/ha, then the consumption of product is about 15,000 t per year.

All kieserite imports in Brazil in 2018 were made through the port of Vila do Conde (Barcarena), Pará state. The current price of kieserite is US$290/t (CFR Vila do Conde port) and US$390/t (FOB fertiliser plant in Barcarena).

19.3.5 Substitute products

The author considers the two main competitors for kieserite in Brazil would be K-Mag and Polysulphate, as both contain magnesium and sulphur, although they also contain other nutrients such as potassium and calcium.

K-Mag

K-Mag is a natural rock fertiliser, obtained from mineral langbeinite imported from the USA by Mosaic; it contains 21% of potash, 21% of sulphur and 10% of magnesium. The content of potash-in-product provides a competitive advantage for this over kieserite.

[9] Cerrado vast tropical savanna ecoregion of Brazil, particularly in the states of Goiás, Mato Grosso do Sul, Mato Grosso, Tocantins and Minas Gerais.

In 2018, the market of K-Mag in Brazil was 43,000 t. In 2019, the outlook is a 10% increase on 2018 volumes. All K-Mag enters Brazil via the ports in the south of Brazil, Santos and Paranagua – Mosaic markets its product mainly in the central south region (Sao Paulo, Minas Gerais, Goias states).

The FOB prices of K-Mag in Sao Paulo and Minas Gerais state varies between US$380/t and US$400/t. In Mato Grosso state, it is reported as US$430/t.

The main crops for K-Mag, according to Mosaic, are coffee (250 to 450 kg/ha), vegetables (200 to 500 kg/ha), and soybean (70 to 120 kg/ha). Mosaic does not supply to the palm oil market in Pará state, because of the cost of logistics.

Mosaic has its own commercial team in Brazil and all marketing is carried out directly to/with the farmers.

Polysulphate

Polysulphate is a natural rock fertiliser, obtained from mineral polyhalite imported from UK by ICL. It contains 14% potash, 19% sulphur, 12% calcium, and 3.6% magnesium. The advantage of polysulphate is that it has four nutrients. The disadvantage is that it has lower magnesium, which reduces its desirability in the palm oil market.

The market for polysulphate in Brazil in 2018 was approximately 90,000 t and the expectation in 2019 is that it will be 130,000 t. It is imported into Rio Grande and Porto Alegre (Rio Grande do Sul state), Santos (Sao Paulo state), Vila do Conde (Pará state) and Itacoatiara (Amazonas state). ICL do not have a sales team in Brazil and all sales are undertaken through distributors. Polysulphate is a new product in Brazil, with product marketing carried out from 2015 onwards.

According to distributors, the crops it is used for are soybean (100–150 kg/ha), corn (70–100 kg/ha), cotton (200–300 kg/ha), sugarcane (150–250 kg/ha), coffee (300–500 kg/ha) and vegetables (200–500 kg/ha). Some is sold to the palm oil market, but this is restricted by the lower magnesium concentration of the product.

The FOB prices of polysulphate varies between US$280/t and US$300/t.

19.3.6 Conclusions and price assumptions

The fertiliser market in Brazil is large. In 2018, 35.6 Mt of fertiliser was sold; of this, 77.5% was imported and 22.5% was manufactured locally, The most likely consumers of the Project's kieserite are the palm oil growers in Pará state, as palm oil trees have a very high demand for both magnesium and sulphur (although it has been demonstrated that coffee and cotton would also benefit from kieserite). The location of the Vermelho plant in the centre of the Pará state gives its distribution a competitive advantage over select imported products. The Project will produce approximately 150,000 t of kieserite a year, which is 10 times the current market for imported kieserite. This means there would be oversupply which would indicate a price reduction and would require substitution of other agro-products for all of the Projects kieserite to be consumed within Brazil. There is potential for the kieserite to be exported via Villa do Conde to international markets. For this study it has been assumed that it would be unlikely for the current prices (approximately U$380/t FOB Barcarena) to be realised. Consequently, HZM has assumed a kieserite price of US$180/t (delivered) – less than half of the current price in Barcarena.

19.4 Offtake agreements and other material contracts

HZM has no offtake agreements in place.

The Company has not entered into any contracts with regards to property development, mining, concentrating, smelting, refining, transportation, product handling and sales for the Project.

HZM will enter into contract negotiations after project financing has been finalised.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Overview of the region

The Project is located 3km from the town of Canaã dos Carajás, founded in 1994, which forms the southern limit of the Carajás Mining District (CMD) Pará state, north of Brazil. The CMD is a host to a number of tier 1 iron, nickel and copper mines operated by Vale

The Carajás Mineral Province was originally discovered by accident when a US Steel helicopter was forced to land on a hill in the area to refuel in 1967, with the first iron ore mine called N4E Mine coming into operation in 1985.

In 2004, Vale started to operate the Sossego Copper Mine after several infrastructure municipality improvements, and most recently (2017) ramped-up the S11D project, one of the largest standalone iron operations in the world. As a result of the advances of mining in the region, there has been a significant influx of people and investment, which has in turn promoted changes and improvements in the areas of economic growth, cultural diversity and a more developed economy than nearby towns, which is heavily centred around mining related activities.

The Canaã dos Carajás Municipality bagan as an agricultural settlement in 1982. The original settlement was around 1,500 people in the 1980s, which increased to approximately 4,000 in the 1990s and after the mining boom impacts were realised, the population increased almost tenfold to approximately 36,000 inhabitants (IBGE, 2017).

20.2 Regulatory requirements

The permitting process for a large-scale mine in Brazil runs along two parallel pathways. These are the mining permit (direitos minerais) and environmental permit (licenciamento ambiental) pathways and each one is managed by separate and independent public authorities. Both mining and environmental permits are necessary to build the Project. The Mineral National Agency (ANM) and the Pará State Environmental Agency (SEMAS) are key agencies relevant to this project for permitting purposes. The environmental permits will mostly be filed with the State Environmental Agency, SEMAS, and not the Federal Environmental Agency, IBAMA, because the project is 100% located within Pará State. According to Resolution no. 237/97, issued by the Brazilian National Environmental Council (whose acronym is CONAMA), three consecutive environmental permits must be requested throughout the LOM, and have been listed below.

20.2.1 Preliminary Licence (LP)

Government approval of social and environmental viability of the Project, with conditions specified for the HZM to implement in order to obtain further licences. The granting of the LP is often regarded as the most important licence as it outlines all the basic parameters of the Project to be accepted by all the parties involved and is the only environmental licensing process that requires COEMA approval, which is a Pará state Environmental Committee consisting of State Government ministers, NGOs and other representatives from society. Further environmental licensing processes always follow the basic parameters stipulated in the LP and are progressed directly with SEMAS.

20.2.2 Installation Licence (LI)

Government approval for the construction of the Project (open pits, erection of process plant and support buildings), with conditions specified for HZM to implement in order to obtain the Operating Licence.

20.2.3 Operating Licence (LO)

Government approval for the continuous operation and production of the mine/plant obtained after construction is complete and all LI conditions met. Additional Environmental Permits necessary to complete the licensing requirements are discussed below.

20.2.4 Other mining permits

The Project will require the following mining permits, in addition to the permits listed above:

- Mineral Exploration licences
- Final Exploration Report approval
- Mine Plan (Plano de Aproveitamento Economico – PAE) approval
- Mine Lease (Land Rights) approval
- Mine Concession
- Concession Ownership ("Imissão de Posse").

20.2.5 Previous permits

By 2016, the following mining and environmental permits were granted to Vale:

- EIA RIMA Studies (Environmental Impact Study (EIS) and Environmental Impact Report (EIR)) issued
- Award of LP
- Environmental Controls Plan issued
- Application for LI
- Final Exploration Report approved
- Mine Plan (Plano de Aproveitamento Economico – PAE) approved.

Whilst a new permit pathway is proposed (see Section 20.2.6); the previously awarded permits for Vermelho are considered a favourable indicator for HZM. SEMAS had previously awarded the LP for Vermelho to Vale to produce 46,000 t/a of nickel, which means that in 2005/2006, the Project was deemed as socially and environmentally viable. As HZM is redesigning Vermelho with a reduced total production (approximately 20,000 t/a of nickel) and with staged production, the social and environmental impacts are expected to be less significant than those originally approved by SEMAS for the Project via the LP.

20.2.6 Proposed permit pathway

HZM will update and complement current Environmental and Social Studies and management plans. Given that the Company is adapting the Project to recent developments in technologies – this will improve engineering layouts. A new licence application will be presented to government agencies for Vermelho.

The permitting will re-commence with a new Environmental Impact Study and Report to Society (EIA RIMA). It is anticipated to commence post-publication of this PFS.

The expected licensing studies and stages are:

- Preliminary Licence (LP):
  - Environmental Impact Study and Report to Society ("Estudo de Impacto Ambiental, e Relatorio de Impacto Ambiental" – EIA RIMA).
- Installation Licence (LI):
  - Environmental Control Plans (Planos de Controle Ambiental – PCAs)

- Vegetation Suppression authorisation
- Fauna removal and monitoring authorisation
- Consent for cave management program (Federal Agency – IBAMA)
- Resettlement and Land Action Plan
- Grants for Water Use:
  - Grant for water catchment to supply industrial plant
  - Grant for water catchment for human consumption
  - Grant for containment dyke
  - Grant for wastewater discharge.

- Operation Licence.

HZM already possesses the Operational Licence for the exploration activities from the Municipal government and has requested Terms of Reference request for the Vermelho EIA RIMA/LP from SEMAS.

20.3 Environment

20.3.1 Previous environmental studies

Important environmental studies for the advancement of Project licensing stages have been performed, such as:

- Speleological survey
- Forest Inventory
- Climate characterisation
- Water availability
- Topographic studies
- Archived integrated EIS and control plans.

HZM will utilise these studies and baseline data collected by previous owners to inform and expedite new EIA RIMA studies.

20.3.2 Proposed new environmental studies

An environmental studies program was presented to the Environmental Agency of Pará state as part of the previous owner's licensing process. As HZM will re-commence the licensing for Vermelho, some studies will be updated and presented again to reflect the current physical, biological and social settings. The main studies to be presented to SEMAS are likely to include an integrated EIA RIMA, with the following items:

- Water availability and quality
- Air and noise baseline
- Soil quality
- Flora and fauna inventory
- Socio-economic considerations
- Resettlement.

In addition to local permit studies, further social and environmental impact assessments/plans will be undertaken in line with International Finance Corporation (IFC) Performance Standards and Equator Principles. The previous studies undertaken demonstrated no intersection with indigenous areas. Brazilian Legislation requires additional Indigenous studies and approvals from the national agency called FUNAI for Projects within 10 km from recognised Indigenous reserves/communities. There are no indigenous reserves within a 50 km radius of the Project.

With respect to vegetation and legally protected areas, Vermelho's influence areas are outside National Forests or Conservation Units. The Project is located within the Amazon Biome and therefore HZM will be required to maintain 80% of any land controlled as a nature reserve (reserva legal). Because decades ago, much of the Project area was deforested due to farming activities, HZM may choose to rehabilitate vegetation in and around the Project infrastructure, which could be assessed as one of the environmental benefits of the Project. HZM will adhere to IFC Performance Standards and ensure a net positive benefit to biodiversity.

20.4 Social considerations

The Project is in a region where mining has promoted significant positive transition in the regional economy over recent years. Economic growth has led to advancements such as new infrastructure, an increase in local businesses, urban population growth and increased consumer confidence.

Such changes have allowed the town of Canaã dos Carajás to be one of the most developed towns in Pará state. The town has one of the best indexes for education (12th in Pará). Canaã dos Carajás and Parauapebas, the neighbouring municipality, hold the second and third highest ranks on GDP per capita in the Pará state. It is important to note that Pará state is one of the least developed states in Brazil with relatively low human development indices in comparison with other states.

Whilst the Canaã dos Carajás municipality has advanced in many human development metrics, the region still has notable poverty levels, with 40% of its population living in poverty, earning below half of the minimal wage (US$125/month, which equates to approximately US$4/day) with improvement required in some health indices, such as child infant mortality rates (78th in the State).

During the 40-year Project LOM, the Company will spend some US$7.4 billion on operations and approximately US$2.8 billion in local, state, and national taxes; this is expected to result in a significant improvement in infrastructure and services, whilst having a limited impact on the existing skilled workforce. A key element for the socio-economic development programs for Vermelho will be to target a reduction in poverty rates through government capacity building, education and training, local supply chain and enterprise development. It is recommended that HZM partner with existing businesses to develop regional solutions to support and improve local health and community agencies.

20.5 Opex and capex estimates for permitting

An Environment, Social, Land, Resettlement and Permit budget estimate of capex and opex was prepared for the Project. Costs were estimated from a variety of sources for the various items considered. The main source of these costs was the Araguaia ferro-nickel project Feasibility Study (HZM, 2018), which included quotes from Environment Resources Management (ERM), Brandt Meio Ambiente and Ramboll consulting groups.

Permit (including social and environmental) costs took into consideration the HZM owners team structure, including:

- HSE Manager
- 4 x HSE specialists (1 = Environment, 1 = Community, 1 = Resettlement, 1= Safety)
- Environment analyst

- Safety analyst
- 3 x HSE technicians
- Site nurse.

Other mining-related costs considered were vegetation clearance, soil removal and relocation of topsoil. Capex and opex estimates were collected for the implementation and ongoing maintenance and reporting to agencies on PCAs.

20.5.1 Closure cost estimate

Mined-out areas will be reclaimed as a continual process as part of mine planning and production. Topsoil stockpiles will be maintained throughout LOM. The remaining material will be stabilised and vegetated on the surface. At the end of the Project, industrial structures will be decommissioned, roads and infrastructure will be removed, and areas will be revegetated, unless an alternate arrangement has been made with authorities. Any areas of the mine site that have not yet been rehabilitated will be made safe with fencing where necessary, to ensure the safety of people and livestock.

HZM's sustainability team estimate a closure cost of BRL371 million, which is in line with mine closure cost estimations for similar sized mines in the region. It is noted however that a mine closure plan needs to be undertaken to meet the specifications of the HPAL mine as part of the EIA RIMA and FS, and this may impact the cost estimates.

20.5.2 Opex (permitting)

Opex are essentially an extension of the PCAs. In general, the opex included the following activities:

- Site management and supervision (HZM owner's team)
- Social, physical and biological PCAs
- Resettlement activities
- All socio-environment costs estimated for infrastructure components of the Project.

Opex costs for permits (no land) continue at approximately BRL2.8 million per annum.

Direct social investment is estimated at BRL0.50 million per annum for the LOM. Additionally, beyond the payback period, the mine will be able to enact "Lei do Rouanet" which is a Brazilian law enabling companies to allocate up to 4% of income tax to social and cultural projects in the region. Direct social investment requirements will be continually re-assessed in parallel with community consultation and social-management systems.

20.5.3 Environmental compensation capex/opex

Environmental compensation is an instrument of public policy. This instrument aims to guarantee to society a compensation for the losses caused to the biodiversity by projects that have environmental impacts. Major mining/infrastructure projects are generally required to pay an Environmental Compensation fee to the environmental licensing agency. The objective of this compensation is to fund the preservation of biomes and/or ecosystems, preferably similar to that in the region of the impact, ensuring the continuity of the country's natural resources through the protection of Conservation Units. The Law No. 9985/2000, sets out the requirements for this compensation.

The state of Pará, applies the federal legislation Decree No. 2033/2009, whereby the agency quantifies the environmental impact of each project and establishes a compensation fee in line with the perceived impact, which varies from 0% to 2%.

The Project environmental compensation was estimated by HZM's sustainability team based on similar projects of size in the region, giving a value of 0.9% to 1.5%. The economic model has assumed the upper limit, a conservative value of 1.5%, which is applied to the total capex of the Project. Therefore, the total Project environmental compensation is estimated at BRL21.6 million to be paid in three instalments over year 1 and year 2 of construction and year 1 of operation.

Land-related values and resettlement programs have been included in the overall capex of the Project and are available for due diligence purposes.

21 CAPITAL AND OPERATING COSTS

All monetary values reported in this section are in US$.

21.1 Cost estimation overview

This study assumes an autoclave feed rate of 1.0 Mt/a after an initial ramp-up period, followed by an increase to 2.0 Mt/a after three years. The intent being that the doubling of capacity in year 3 is funded by cash flow. The ore processing methodology is the hydro-metallurgical conversion of a nickel and cobalt bearing laterite ore into nickel and cobalt sulphates using the HPAL process. A saleable by-product of the selected process route is kieserite ($MgSO_4 \bullet H_2O$) which could be sold into the Brazilian fertiliser market or exported. The nickel and cobalt sulphate products are assumed to be bagged at the plant site transported by road to Vila do Conde Port for sale to customers FOB. The kieserite is assumed to be bagged on site and transported by truck to buyers or agents' depots.

The two years of pre-production capex in the cash flow model have been allocated 30% in year 1, 30% in year 2, and 40% in year 3. 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.

21.2 Capital expenditure

21.2.1 Basis of estimate

The estimate is based on the AACE Class 4 standard, with an estimated accuracy range between -25% and +20% (Table 21-1) of the final Project cost (excluding contingency).

The process plant capex estimate is based on the 3D layout for all areas of the Project and is supported by mechanical equipment lists and engineering drawings from which material takeoffs have been calculated. The costs for these items have been derived from vendor quotes for the equipment and materials. The process plant capex estimate is before tax (duties and taxes calculated separately), includes contingency, but excludes escalation. The process plant capex occurs in two separate events, initial (Train 1) and Train 2 (Stage 2) where production is doubled in year 3.

Table 21-1 Cost estimate classification

Estimate class	Engineering definition of accuracy	End usage	Methodology	Expected accuracy range
Class 5	0% to 2%	Concept screening	Capacity factored, parametric models, judgment, or analogy	L: -20% to -50% H: +30% to +100%
Class 4	1% to 15%	Study or feasibility	Equipment factored or parametric models	L: -15% to -30% H: +20% to +50%
Class 3	10% to 40%	Budget authorisation or control	Semi-detailed unit costs with assembly level line items	L: -10% to -20% H: +10% to +0%
Class 2	30% to 75%	Control or bid/tender	Detailed unit cost with forced detailed takeoff	L: -5% to - 15% H: +5% to +20%
Class 1	65% to 100%	Check estimate or bid/tender	Detailed unit cost with detailed takeoff	L: -3% to -10% H: +3% to +15%

All costs are in 2019 US$ terms. Where the original estimated currency is other than US$, the rates used are shown in Table 21-2.

Table 21-2 Exchange rate and proportion of estimate

Currency	Exchange rate (1 US$)
BRL	3.80
CAD	1.34
CNY	6.71
EUR	0.88
US$	1.00

No allowances are made for hedging of foreign currency variability between estimate date and the settlement of the order.

The capex estimate includes all the direct and indirect costs, local taxes and duties and appropriate contingencies for the facilities required to bring the Project into production, as defined by a feasibility level engineering study. The estimate is based on an EPCM implementation approach and the Project contracting strategy outlined in Section 24.1.

21.2.2 Capex estimate assumptions

The following assumptions are made in preparing the estimate:

- Required statutory permits are in place according to the scheduled milestones.
- EPCM works are completed in accordance with the Project schedule.
- Site access is granted, as per the timeline shown in the Project schedule.
- Land acquisition and right-of-way have been established for construction.
- Weather conditions are not at extremes that may disrupt the continuance of safe work. A nominal allowance for severe weather events is made in the labour productivity assessment, based on the contractor's proposal which are included in the informed unit costs/rates.
- Project delivery will not be constrained because of concurrent projects.
- Suitable fabrication shops are available locally or identified by contractors during their bidding process.
- Special cranes and special freight services, as required for heavy/difficult lifts or oversized goods for transport, are available locally.
- Pre-commissioning check-outs prior to mechanical completion are included.
- Mobile equipment, as well as temporary and standby power generation equipment, is to be leased and not purchased.

21.2.3 Capex estimate exclusions

The following items are excluded from the capex:

- Any scope variations and deferred capex (addressed with sustaining capital)
- Changes to industrial relations laws; and extended periods of industrial unrest
- Finance and interest charges for Project duration
- Any environmental requirement not identified in this estimate
- Extreme weather conditions of significant duration
- Soil remediation for any in-situ hazardous contaminants
- Cost of delays associated with obtaining statutory approvals (e.g. building or development approval)
- Sunk costs (e.g. cost of this study and previous ones, etc.)

- Market forces related to the imbalance of supply and demand beyond the expected annual rate of inflation of prime commodities, such as steel, copper and pipe
- Effect of related concurrent projects on the availability of construction labour and materials
- Fuel price and foreign exchange variations
- Acts of God.

21.2.4 Process plant capital cost

The plant capital cost estimate is summarised in Table 21-3 with a base date of April 2019 and with no provision for forward escalation. There are two phases to the process plant capex; initial capex with stage 1 construction which plans to deliver 1 Mt/a of autoclave feed, and stage 2 which delivers an additional 1 Mt/a of autoclave feed in year 3 (creating plant capacity of 2 Mt/a of autoclave feed).

The process plant capital cost includes direct costs for the construction of the process plant, including the mechanical equipment, piping and valving, electrical and instrumentation, structural, civils and earthworks. Allowances for the following additional direct costs are applied as a percentage of the installed mechanical equipment costs:

- Painting and protective coatings
- Sustaining capital
- Site buildings
- Mobile construction equipment
- First fills
- Critical spares.

The following indirect costs for the process plant are also included in the estimate as factors of the installed mechanical equipment cost:

- Site temporary facilities
- Mobilisation and demobilisation
- Freight
- Site commissioning and vendor representatives
- EPCM
- PCM.

Table 21-3 Summary of plant capital cost (Simulus)

WBS level 1	WBS description	Stage 1 (US$ M)	Stage 2 (US$ M)	Total (US$ M)
000	Main building/sitewide costs	38.1	0.0	38.1
100	Feed preparation	22.0	22.0	43.9
200	Leach	77.9	77.9	155.7
400	Nickel and cobalt recovery	50.6	7.0	57.6
500	Wastewater	24.6	24.6	49.2
600	Reagents	115.8	115.8	231.5
700	Water	10.7	10.7	21.3
800	Utilities	9.5	0.3	9.8
	Other direct costs	33.6	39.8	73.4
	Subtotal direct capex	**382.7**	**297.9**	**680.6**
	Subtotal indirect capex	**96.5**	**74.3**	**170.9**
	Contingency	95.8	74.4	170.3
	Total plant capex (Train 1 & 2)	**575.1**	**446.7**	**1,021.7**

21.2.5 Mining capital

Mining related capex for the Project are presented in two broad categories: equipment and infrastructure for mining operations (mine capital equipment) and capital development for the establishment of operations. In addition, there will be opex incurred during the pre-production period that will contribute to working capital. Mining capex is summarised in Table 21-4.

Table 21-4 Summary of pre-production mining capital cost

Description	Pre-production (US$ M)
Mine development capital	6.01
Pre-production opex contributing to working capital*	4.77
Total mining capex	**10.78**

**Covers only the costs prior to the first production period. It does not allow for initial lag in revenue from product delivery and sale which may further increase working capital.*

21.2.6 Other capital

Infrastructure

River water extraction

As discussed in Section 1.1, Miptec undertook a detailed design and cost estimate for the Araras River water extraction pumping and pipework to supply make-up water to the plant. The cost estimate developed by Miptec is detailed in Table 21-5 below.

Table 21-5 River water extraction capital cost estimate (Miptec)

Item	Cost (BRL M)	Cost (US$ M)
Preliminary services	0.32	0.09
Earthworks	0.05	0.01
Concrete	1.40	0.37
Steel structure	0.07	0.02
Platework	0.02	0.01
Mechanical equipment	0.27	0.07
Piping and accessories	1.15	0.30
Electrical equipment	0.11	0.03
Electrical material	0.33	0.09
Automation and control	0.13	0.04
Electro-mechanical assembly	2.00	0.53
Subtotal direct costs	**5.87**	**1.55**
Indirects	0.92	0.24
Contingency (31% of direct costs)	2.10	0.55
Total	**8.89**	**2.34**

Mine residue storage facility and sediment control dam

WALM developed a scheme to safely store the beneficiation rejects, and dry tailings developed by the plant for the mine life along with a dam to control sediment from the facilities runoff. These designs are discussed in detail in Section 18.4 and Section 18.5. WALM calculated the initial capital cost of these facilities, as shown in Table 21-6 below.

Table 21-6 Mine residue and sediment control facility capital cost estimate (WALM)

Item	Cost (BRL M)	Cost (US$ M)
Temporary construction facilities/final facilities	0.60	0.16
Preliminary services	1.77	0.47
Recharge drain	10.95	2.88
Internal drainage	4.53	1.19
Waterproofing system	40.56	10.67
Flow detection system	27.48	7.23
Sediment containment dam	5.78	1.52
Total	**91.67**	**24.12**

Energy supply

SM&A was commissioned by HZM to develop an energy supply solution for the Project. This is detailed in Section 18.6. A breakdown of the SM&A capital cost estimate is provided in Table 21-7 below.

Table 21-7 Energy supply capital cost estimate (SM&A)

Item	Cost (BRL M)	Cost (US$ M)
Bay at SE-Integradora	6.02	1.58
15 km powerline to Project	11.64	3.06
Substation and transformers	26.59	7.00
Step-down transformers	4.66	1.23
Subtotal	**48.91**	**12.87**
Contingency (10%)	4.89	1.29
Total	**53.80**	**14.16**

Social, environment and permitting

HZM's underlying assumption(s) with respect to permitting and land acquisition are set out in Section 20. The cost of permitting and land acquisition for the Project has been estimated by HZM and is shown in Table 21-8.

Table 21-8 Estimate of initial permitting and land acquisition costs (HZM)

Item	Cost (BRL M)	Cost (US$ M)
Vermelho Main Permit and Control Plan	35.50	9.342
Other Vermelho Permits	0.91	0.239
Transmission Line Project	2.63	0.691
Land Acquisition and Land Compensation	49.07	12.913
Total	**88.11**	**23.186**

Summary of other initial capital

Other initial capital includes infrastructure such as powerline, mine residue storage and settling, road upgrades, and the river water extraction pipeline along with other land acquisition and permitting. These costs are summarised in Table 21-9.

Table 21-9 Summary of other initial capital

Item	Cost (BRL M)	Cost (US$ M)
Tailings and sediment	91.67	24.12
Pumping	8.89	2.34
Powerline	53.80	14.16
Road	9.85	2.59
Permitting and land acquisition	88.11	23.19
Total	**252.31**	**66.40**

21.2.7 High-pressure acid leach plant initial capital summary

A summary of the process plant initial capex estimate is shown in Table 21-10. The initial capital includes the first of two HPAL trains, plus the refinery and associated reagents and utilities process areas. The process plant operates with only one HPAL train for the first three years of operation, with the second coming online in year 4.

Table 21-10 Summary of plant initial capital cost (Stage 1)

WBS level 1	WBS description	Initial capital cost (US$ M)
000	Main building/sitewide costs	38.1
100	Feed preparation	22.0
200	Leach	77.9
400	Nickel and cobalt recovery	50.6
500	Wastewater	24.6
600	Reagents	115.8
700	Water	10.7
800	Utilities	9.5
	Other direct costs	33.6
	Subtotal direct capex	**382.7**
	Subtotal indirect capex	**96.5**
	Contingency	95.8
	Total plant capex (Train 1)	**575.1**

21.2.8 Sustaining capital

The LOM sustaining capital requirement of Vermelho is estimated to be US$66.4 million, comprising contract mining costs associated with mine development, contractor mobilisation and demobilisation every five years. An annual allowance of 2.5% of the total direct costs have been applied to calculate the sustaining capital for the process plant and has also been included in Table 21-11.

Table 21-11 Project sustaining and closure capex summary

Area name	Timing	Amount (US$ M)	Responsible
Ongoing mine development and contractor replacement	Annual average	0.71	HZM
Process plant (annual rate)	Annual	17.0	Simulus
Land acquisition (year 2)	Year 2	1.33	HZM
Mine and plant closure capex	End of mine	29.4	HZM

21.2.9 Basis of quantities

Engineering drawings of the plant were prepared by Simulus, which include detail on:

- The location and size of mechanical equipment
- Critical piping between major mechanical equipment items
- Primary steel beams and columns including major pipe-racks, access platforms and stair towers
- Major electrical equipment locations.

The balance of quantities is calculated based on layouts and general arrangements or factored based on engineering judgement. All quantities exclude allowances, but have incorporated civil, structural, architectural, mechanical, piping, electrical and instrumentation and control aspects.

Procurement

Firm or budgetary prices were obtained for majority of packages, with the balance estimated in-house. Ex-works pricing was sought for the major mechanical equipment packages using competitive pricing submissions from equipment suppliers. Where budget quotes were not obtained, in-house historical pricing and estimates were used.

Mining costs

The capitalised mining costs represent contract mining costs associated with the initial stages of development of the mine site.

Direct field labour

The prefabrication and installation costs are based on budgetary data obtained from local contractors. Labour costs were provided per unit material takeoff (MTO) quantity. As such, labour rates were not directly used for calculating labour costs. Labour hours were provided by the contractors in the form of histograms and validated against in-house data. This information was used to determine the size of the construction accommodations and others indirect costs such as meals supply.

Contractors' distributables

Contractors' distributables are included in their respective crew rates. These rates cover construction equipment and expenses to support and deploy installation labour. Cost components covered by these rates include mobilisation and demobilisation, construction facilities, construction equipment, material transportation from warehouse to job site, construction supervision support, manual indirects, home office costs and contractors' fees.

Labour productivity

The direct field labour hours are based on information received from local suppliers in the form of histograms. As such, the labour productivity is 3.0. This information was used to size the construction camp accommodations. Operational labour will be sourced from local municipalities.

Freight costs

International freight costs (plus insurance) were requested from all suppliers as delivered duty paid (DDP) to the Port of São Luis or Vila do Conde (Barcarena). Where international freight costs were not available, they are calculated at 6.5% of the equipment/materials costs. The domestic freight portion of the international items (between port of Vila do Conde and the Project site) is estimated at 2% of the equipment/materials costs. International shipping insurance is estimated at 0.3% of the equipment/materials costs.

The domestic freight costs (between the state of origin and the Project site) for those items that are supplied within Brazil are estimated at 4.8% of the equipment/materials costs (equipment and piping), for the other bulks, freight is estimated at 2% of the material cost.

21.2.10 Customs duties and taxes

The applicable local taxes, customs duties and fees for all equipment, materials and services are calculated by a Brazilian third-party tax specialist (L&M Assessoria Empresaria), retained by HZM and added to the estimate. A study on the taxation basis for calculation of the initial and sustaining capex taxes was undertaken.

The following is noted:

- EPCM services costs. EPCM services are estimated based on a build-up of man-hours using a deliverables-based approach, organisation charts, and the EPCM schedule (excluding taxes and escalation). Provision was made in the EPCM estimate for pre-operational testing and pre-commissioning punch lists to mechanical completion. This includes construction crews, commissioning manager, field commissioning engineers, field planners and post-handover personnel. Provision has also been included for commission and start-up assistance.
- Spare parts. Major mechanical spares for construction and commissioning have been identified from the vendor returned quotes where provided. To supplement the major spares list, factors of 1.5% for commissioning spares and 3.0% for operational spares have been taken from all the mechanical and electrical equipment not represented on the major spares list.
- Vendors representatives on site. Vendors representatives will be required to supervise installation and start-up of certain equipment. Costs for vendors representatives for construction and commissioning have been allowed for as factors from the supply cost of the mechanical and electrical equipment – construction vendors at 2.5% of equipment supply cost and commissioning vendors at 2.0% of equipment supply costs (excluding taxes, freight and escalation).

21.2.11 Owners costs

The following owners' costs have been allowed for:

- Owner's team: This is the client team, responsible for execution of the Project and includes the Project management, operational readiness, commissioning, and the performance testing teams. It excludes the mine/processing plant operations team, which will be accounted for under opex. This portion of the cost is included in the EPCM integrated team costs.
- Communication: All communication activities including internal and external disclosure, institutional material, training, internal communication vehicles, press, audio-visual records, etc.
- Health and safety (including security): All services required for the deployment and operation of the Project, including risk control, accident prevention, continuous improvement, loss prevention and security.
- Administration: Funds allocated to corporate areas that are borne by the Project.
- Insurance (excluding freight): Insurance costs for engineering and construction activities and civil liability.
- Pre-operational expenses: The operating expenses until the end of the commissioning and performance testing periods, including the pre-operations team, first fills, power, fuel, water, etc.
- Environmental: All costs associated with environmental studies, assessment, compensatory measures and remediation of environmental liabilities.
- Community: Costs associated with community engagement activities such as social impact management, social investment, social dialogue, etc.
- Sustainability. All services related to sustainability that are not included in the environment, community relations or communication; in particular, investments for carrying out voluntary actions.

21.2.12 Closure costs

Mined-out areas will be reclaimed as a continual process as part of mine planning and production. Topsoil stockpiles will be maintained throughout LOM. At the end of the Project, industrial structures will be decommissioned, roads and infrastructure will be removed, and areas will be revegetated, unless an alternate arrangement has been made with authorities. The mine site will be made safe with fencing where necessary, to ensure the safety of people and livestock. A water retention dam will remain after mine closure for containment of any seepage water from the rejects/dry stacked tailings disposal facility. Included in the cost assumption of the closure is the estimated salvage value based on the tonnes of metal and prevailing scrap metal price.

21.2.13 Contingency

Contingency is a provision for unforeseen or inestimable costs within the defined Project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy and applied to provide an overall level of confidence in costs and schedule outcomes (in this case, targeting an 50% confidence level). The contingency is meant to cover events or incidents that occur during the course of the Project which cannot be quantified during the estimate preparation and does not include any allowance for Project risk.

Contingency does not cover scope changes, force majeure, adverse weather conditions, changes in government policies, currency fluctuations, escalation and other Project risks. The recommended amount for contingency was developed using a probabilistic model based on the quality of the information and level of engineering.

The total contingency carried in the initial capex estimate is US$97.7 million, the majority of which lies with the HPAL plant (US$95.8 million) which represents 25% of the plant direct costs.

21.2.14 Capital cost summary

A summary of the calculated capital costs for the Project is provided in Table 21-12 below.

Table 21-12 Capital cost summary (including contingency)

Capital cost component	Initial (Stage 1) (US$ M)	Stage 2 (year 3) (US$ M)	Remainder (US$ M)	Total LOM (US$ M)
Process plant	575	447		1,022
Mining pre-production	11			11
Tailings and sediment	24			24
Pumping	2			2
Powerline	14			14
Road	3			3
Permitting and land acquisition	23			23
Mining sustaining			22	22
Other sustaining (including land permitting and land)			1	1
Closure			29	29
Total capital cost	**652**	**447**	**52**	**1,151**

21.3 Operating expenditure

Opex for the Project have been prepared based on the Project physicals, detailed estimates of the consumption of key consumables based on those physicals, and the unit cost of consumables. The estimate is supported by detailed engineering, benchmarking and market pricing of key consumables and costs. Royalties and taxes are calculated based on the advice of the HZM Brazilian mining tax advisor, L&M Assessoria (L&M).

All financial values in this section are in US$ unless otherwise stated. Brazilian Real (BRL) is shown as R$. A summary of the opex is provided in Table 21-17.

Opex is comprised of physicals, labour, reagents and operating consumables, freight and power costs, mobile equipment, utilities, maintenance and mining contract costs, external contractor costs, environmental, and miscellaneous/other General and Administrative (G&A) costs.

Opex estimates were prepared or advised by the groups discussed in Section 21.1.

No contingency was applied to the opex.

21.3.1 Exclusions

The following items are not included in the opex 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
- Foreign currencies exchange rate fluctuations although these have been reviewed in the sensitivity analysis
- Forward escalation beyond the estimate base date of (October 2018) to the Project completion
- Risks due to potential government policy changes, labour disputes or permitting delays.

21.3.2 Physicals

Physicals for the opex estimate were derived from the mining schedule, and heat and mass balances developed from the Basis of Design and the Process Design Criteria.

21.3.3 Labour

The organisation charts and manpower cost studies have been completed. The manpower costs include a taxes multiplier of 1.77 over the base salary to include all local taxes payable. The organisation structure for the Project is divided into three main groups, namely management, operational (mines and geology, plant and plant maintenance) and supporting groups (safety, health, environmental, security, medical, IT, accounting, procurement, warehousing, general administration, marketing, sales, external relations).

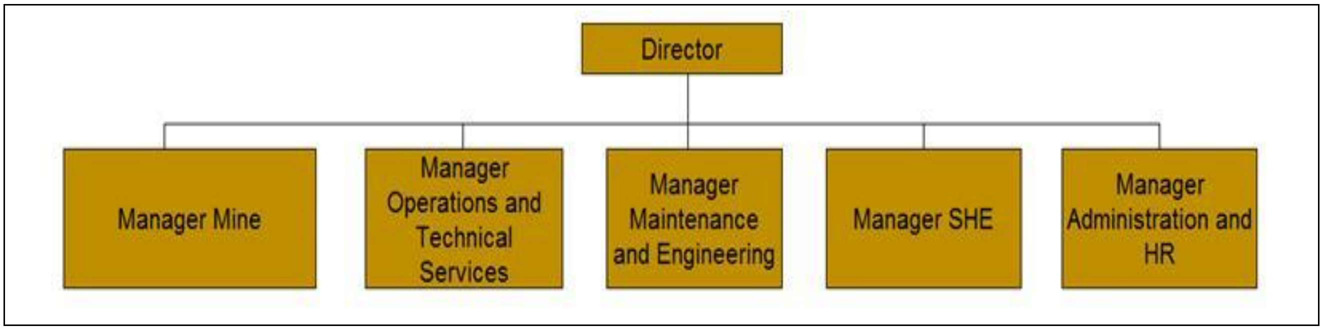
The normal work week in Brazil is 44 hours. One hour for lunch per day is not included in the 44 hours. For 24-hour coverage, there will be four teams working on 8.5-hour shifts. Three teams working and one on leave. This arrangement will allow an overlap period of 30 minutes in each shift change to ensure proper operational exchange at the various work positions.

The annual manpower costs are summarised by department in Table 21-13. These costs are total costs and include tax multipliers. A high-level Project organogram for management is shown in Figure 21-1.

Table 21-13 Annual manpower costs by major departments

Departments	No. of people	Annual manual costs	
		BRL M/a	US$ M/a
Management	4	1.09	0.31
Mining (owners' personnel)	18	2.21	0.63
Plant management and technical	10	1.86	0.49
Plant operations	85	6.70	1.76
Plant maintenance	48	6.32	1.66
HSE + Admin departments	28	2.19	0.63
Total	**193**	**20.38**	**5.49**

Figure 21-1 High level Project organogram for management



Source: Simulus, 2019

21.3.4 Mining operating cost

All mining for the Project will be undertaken by the contractor (which is the standard practice in Brazil). A request for budget quotation (RFQ) was issued to a select group of earthmoving contractors for HZM Araguaia project in 2017. Responses to the RFQ were received from three contractors operating in-country. Unit variable rates from the contractor submissions were used to develop a mining cost estimate based on the final LOM schedule. The proposals included detailed costs for haul type (grade and length, loaded and unloaded etc.).

These costs, which were considered accurate in 2017, were escalated using the IPCA (Brazil broad consumer price index) of 2.21% for 2017 and 3.75% for 2018 and applied to each haul identified in the schedule to arrive at an updated mining cost. The costs were then benchmarked against other operations. The estimate also included mobilisation and demobilisation every five years to account for a contractor change. Drill and blast costs (which were not part of the Araguaia scope) were developed from operating mine data in Brazil and applied to Vermelho. Table 21-14 summarises the mining contractor costs used in the PFS.

Table 21-14 Summary of mining contractor cost estimate

Cost area	LOM cost (US$ M)	Unit opex (US$/t ore)	Unit opex (US$/t Ni)
Mobilisation and site establishment*	2.50	0.018	2.71
Demobilise and restore site	1.57	0.011	1.70
Site preparation	27.12	0.192	29.35
Load and haul to ROM or stockpile	562.52	3.981	608.82
Load and haul from stockpile to ROM	220.29	1.559	238.42
ROM operations	141.83	1.004	153.50
Drilling for blasting	13.11	0.093	14.19
Blasting	6.22	0.044	6.73
Grade control (drill, sample and assay)	21.33	0.151	23.08
Dayworks	15.51	0.110	16.79
Beneficiation Rejects hauling	154.67	1.094	167.40
Tailings hauling	221.74	1.569	239.99
Total	**1,388.41**	**9.825**	**1,502.68**

**Allocated to sustaining capital*

The following is noted:

- Grade control drilling. Grade control drilling will be carried out using RC drills. The hole spacing will be 12.5 m x 12.5 m and will be drilled from the surface. The estimated LOM drilling will be 614,000 m, and average 36,000 m per year when mining is carried out (first 17 years).
- Drill and blast. This was estimated using the silica content of the material being mined the low silica material is assumed to be free dig, it is assumed that 25% of the medium silica material will require blasting; and it is assumed that all of the high silica material will require blasting. The average powder factor required to achieve acceptable fragmentation is estimated to be 0.20 kg/m^3 for the volume that is blasted.
- Load and haul costs. Haul profiles were calculated for each period, material type, source and destination combination. Each segment height change was then converted to an equivalent haul comprising 1:10 gradient distance with the remaining distance as a horizontal haul.
- ROM operating costs. ROM operations account for loading and hauling ore from the ROM pad stockpile to the crusher.
- Tailings and rejects load and haul. The costs of load and haul of tailings and rejects from the processing plant to the storage facility (about 4.5 km).
- Waste dump rehabilitation. The consolidated cost of rehabilitating the waste dumps was estimated to be US$0.76/bcm of waste mined, with a total rehabilitation cost of US$9.28 million, which includes topsoil loading for the final formation, and rip and seeding of waste dump top and sides. No allowance was made for the ongoing monitoring and management of the landform once it was completed in the mining costs. These costs are included in the project closure costs (Section 21.2.12).
- Dayworks. A nominal provision for daywork was included in the estimate. Dayworks will cover for miscellaneous works not specified in the scope that will arise from time to time during the normal course of operations.
- Mobilisation and demobilisation. A provision is made for these costs every five years to accommodate the costs associated with a change of mining contractor.

21.3.5 Process plant operating cost (Simulus)

An opex estimate was developed for the plant based on the consumption rate of key commodities and reagents, derived from the energy and mass balances along with equipment manufacturers' recommendations and industry best-practice. The cost of these commodities and reagents, are based on written quotes from suppliers or marketing companies (Table 21-15).

Table 21-15 Plant operating expenditure summary*

Description	Opex per annum (US$ M)	Unit opex (US$/t Ni)
Directs		
Labour (excluding admin and environmental)	4.0	114.4
Power	7.1	202.5
Reagents	102.6	2,942.3
Maintenance consumables	36.4	1,039.8
Subtotal – Direct Costs	**150.1**	**4,299.0**
Indirects		
Process Plant general and administration	0.1	2.7
Freight costs (product to market)	24.2	692.5
Major shutdown costs	0.5	14.3
Subtotal – Indirects	**24.8**	**709.6**
Total Process Plant	**174.9**	**5,008.5**

**These costs are estimated on year 6 in the mining schedule which is the year when the most nickel is produced. Other years will have different cost and the LOM average will be different.*

Process plant labour

In addition to plant operations, process plant labour includes plant management and technical; maintenance labour; occupational health and safety; and training.

Process plant general and administration

The process plant general and administrative costs include allowances for plant mobile equipment maintenance and fuel; miscellaneous equipment hire; and safety equipment.

Product transport

Product transport in from site to final destination is estimated as BRL347/t dry product for road transport plus US$55/t for sea-freight, for a total annual cost of US$24.2 million.

Major shutdown cost

An annual allowance of US$500,000 is included for a major plant shutdown, corresponding with the biennial sulphuric acid plant shutdown(s) for the replacement of converter tower catalyst. The entire process plant will be shut down during this period for major maintenance.

Power consumption

Based on the process plant electrical load list and the planned utilisation of the loads the calculated power consumption per year 297.7 GWh, inclusive of an allowance for ancillaries (small power/lighting etc.). The sulphuric acid plant's steam-turbine generator generates 170.8 GWh per year, for a net shortfall of 127.0 GWh which is supplied by the grid. The total installed power is 60.5 MW.

HZM commissioned a power cost study to review power generation options. This was prepared by the Brazilian electrical power systems consultancy, Enercel. The result of the study is that the long-term power cost for the Project is predicted to be BRL204.3/MWh for consumption. Separately, a study by SM&A was commissioned by HZM to estimate the capital costs of power supply. A monthly supply cost of BRL4,406/MW installed was estimated in this report (including 40% taxes). On this basis, the annual cost for grid supplied power is BRL26.8 million, or US$6.93 million.

Maintenance and consumables

Maintenance was calculated based on a percentage of the installed equipment cost and is intended to cover the cost of all maintenance items required for the process plant. Mining equipment maintenance is not part of this element and is included in the contract mining rate. The maintenance cost covers regular maintenance on equipment, spare parts and replacement cost, lubrication, planned shutdowns and emergency repairs.

21.3.6 Other operating costs

The operating cost estimate includes other costs not related to labour, mining or processing. These costs are summarised in Table 21-16 below.

Table 21-16 Other operating costs

Cost area	Average annual cost (US$ M/year)
External contractors	1.00
Environment social and permitting	0.75
Vehicles	0.49
Miscellaneous	2.09
Residue placement and compaction	0.99
Total	**5.32**

External contractor costs (HZM)

External contractor costs include all contracts, except mining. It would include security, cleaning services, diesel and gasoline fuel station, conveyor maintenance, a canteen and garbage collection.

Environmental, social and permitting

This item comprises the annual cost of environmental assessments and monitoring (air, water, waste, noise, sedimentation and ecology) and was included in the overhead costs. Associated site management and supervision (HZM owner’s team); social, physical and biological PCAs; all socio-environment costs for the Project are included in this cost item.

Vehicles

This is the cost of owning and leasing the vehicle fleet (excluding mining contractor vehicles which also cover ROM operations and reject/tails handling). The owners fleet consists of 29 light and three medium vehicles.

Miscellaneous and other general and administration

This includes miscellaneous and other costs, for example miscellaneous processing/operation costs; access road maintenance, administration and other costs such as insurance, IT, management and technical assistance fees, legal and other fees, government charges, community relations, community development, indirect taxes and royalties, consultants, software licences, travel expenses, etc.

Compaction and placement of mine residue

As described in Section 18.4, the beneficiation rejects, and the dry process residue is trucked to a storage facility and compacted. The cost of compaction and placement is captured in Table 21-16 and excludes hauling which is included in the mining contractor costs.

21.3.7 Royalties

In terms of royalties, the following is noted:

Compensation for Exploitation of Mineral Resources

The Compensation for Exploitation of Mineral Resources (CFEM) is payable by legal entities in the mining industry that exploit or extract mineral resources and is payable upon sale of the mining product. The CFEM rate is usually applied to the entity's net revenues and varies in accordance with the mining product, but cannot exceed 3% of gross revenues, after deducting insurance, tax and transportation costs, paid annually.

For nickel, the applicable CFEM rate is 2% although when combined with the Landowner rights of 1% a rate of 3% was applied throughout LOM. In the case of the Project, the calculation of the CFEM includes the following costs: mining, stockpiling, crushing, administration, maintenance and some environmental costs up to and including calcining. The sum of these costs gives a deductable value that will be multiplied by 3%.

Landowner rights of participation. Mining entities are also required to pay financial compensation to landowners of 50% of the CFEM. Therefore, the total CFEM cost to be borne by the Project is 3% (2% + 1%), equivalent to a LOM royalty of US$22.82 million (US$620,000 per year).

Vale

As part of HZM acquisition of the Project, HZM is obliged to pay Vale a royalty based on 1% of the value of nickel produced in a year capped at 15,000 t of nickel and reducing to 0.5% after 10 years of production.

21.3.8 Operating cost summary

The operating cost estimate is based on an underlying US$1:BRL3.8 exchange rate. The operating cost summary is shown in Table 21-17.

Table 21-17 Operating cost estimate summary (pre-tax)

Area	LOM total (US$ M)	US$/t nickel*	US$/t ore	Average annual (US$ M)*
Mining	981	1,062	6.94	25.81
Mine reside storage	414	448	2.93	10.89
Processing costs	5,785	6,261	40.93	152.23
Royalties (CFEM)	23	25	0.16	0.60
Royalty (Vale)	66	72	0.47	1.74
G&A and other costs	215	233	1.52	5.67
HSE	24	26	0.17	0.63
Total	**7,508**	**8,126**	**53.13**	**197.57**

**Costs shown here represent an average over the LOM; actual costs for these components vary from year-to-year depending on the physicals.*

21.3.9 Taxation

The tax treatment adopted in the financial evaluation was based upon advice taken from L&M. Import duties, withholding and purchase taxes are based on advice from L&M who advised that certain available exemptions, indirect taxation incentives and taxation credits be applied to the cash flow model.

Most direct taxation incentives apply to new investments and are offered by Federal, State or Municipal governments. These generally include substantial reductions in taxes (mainly State value-added tax (VAT) and corporation tax), utility charges and other expenses. ICMS or “Imposto sobre operações relativas à circulação de mercadorias e sobre prestações de serviços de transporte interestadual e intermunicipal e de comunicações” is the State VAT referred to above. Federal incentives (generally income tax reductions) are available for investments in less developed areas such as where the Project is located.

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.

The total tax paid over the LOM is calculated at US$3,554 million. Summarised below are the principal taxes relevant to the Project.

Federal corporate income taxes

There are two income taxes – the corporate income tax and the social contribution tax on profits (which 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, a small nominal amount; currently US$68,500.
- 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 the Superintendência do Desenvolvimento do Amazônia – the Development Agency for the Amazonia Region (SUDAM), valid for the life of the Project. 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 duration of the Project. The corporation taxation regime applied in the cash flow model allows for a taxation rate of 15.25% of the taxable income.

Gross revenues and import taxes

Current Program of Social Integration (PIS) and Contribution for the Financing of Social Security (COFINS) are federal taxes charged on gross revenues, on a monthly basis, under two regimes: cumulative and non-cumulative.

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. PIS and COFINS are generally imposed on the import of goods and services at a combined rate of 9.25%.

Indirect taxes

Both the Federal and State governments impose 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. This means that a tax-effective corporate structure needs to be established prior to construction.

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 VAT (ICMS) is levied on the import of goods and on the movement out of imported and manufactured goods, even if between branches of the same 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.

Application of indirect taxes – capital and operating expenses

Where benefits and special regimes exist for indirect taxes, it was assumed that these are applicable to the Project. It was assumed in this FS that any credits built up on input taxes are offset against taxes on outputs.

Other taxes relevant to the Project

It is assumed that most of the 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 was 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 – the Mercado Común del Sur or Southern Common Market) and provided that the goods have certificate of origin with an also 0% taxation in the country of origin.

22 ECONOMIC ANALYSIS

The evaluation of the Project economics considers all relevant costs and revenues association with the development and execution of the Project. These are used to derive a set of industry standard measures of economic performance. The Project NPV is re-calculated for a range of values for some of the key inputs so that the sensitivity of the NPV can be assessed.

As discussed in Section 21.3.9, taxation and royalties in Brazil are levied at several levels and several incentives are in place to encourage project development. To ensure that the taxation and royalty basis of the economic analysis was applied appropriately HZM secured the services of L&M to review the economic model and apply taxes and tax incentives. The outcomes of their review have been incorporated into the results reported herein.

All monetary values in this section are reported in US$.

22.1 Project economic headline results

Table 22-1 and Table 22-2 show the Project headline economic results before and after taxation for a flat nickel price of:

- Base Case (consensus):
  - US$16,400/t for nickel + US$34,000/t for cobalt + US$100/t net revenue for kieserite.
- WM Long Term:
  - US$19,800/t for nickel+ US$34,000/t for cobalt + US$100/t net revenue for kieserite. The WM long-term price.

Table 22-1 Project economic model headline results before taxation

Item	Unit	Nickel price basis (US$/t Ni) *	
		Base Case (consensus) 16,400	WM Long Term 19,800
Net cash flow	US$ M	10,379	14,655
NPV_8	US$ M	2,342	3,185
IRR	%	28.8%	34.5%

**Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t of the by-product, kieserite*

Table 22-2 Project economic model headline results after taxation

Item	Unit	Nickel price basis (US$/t Ni) *	
		Base Case (consensus) 16,400	WM Long Term 19,800
Net cash flow	US$ M	7,304	9,546
NPV_8	US$ M	1,722	2,373
IRR	%	26.3%	31.5%

**US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t of the by-product, kieserite.*

22.2 General criteria

HZM prepared a cash flow and financial analysis model ("the model") based on inputs derived from mining and processing schedules reflecting the PFS capital and opex estimates including applicable royalties (federal and state) for the Project. The mining schedule used for the model (which drove processing and revenue) was developed on a yearly basis; the construction schedule is also modelled on an annual basis.

The model was based on the following:

- 100% equity ownership by HZM
- Costing from April 2019
- 30-month pre-production period for plant construction
- No cost escalation,
- All costs reported in US$ and where costs were estimated in Brazilian Reais the exchange rate used was 3.8 BRL to the US$.

The objective of preparing the cash flow model was to:

- Collate all the inputs for the following disciplines into a single model:
  - Mining, processing, metallurgical and metal pricing
  - Pre-production capex
  - Production sustaining capital
  - Opex, environmental costs and social costs
  - Rehabilitation, salvage and closure costs
  - Royalties (CFEM, Vale and state)
  - Taxation, and taxation incentives.
- Be sufficiently flexible to enable options (capital and operating configurations) to be evaluated.
- Provide sufficient information to management so that they are supported in any decision-making process.
- Provide the basis for future studies and a decision whether to proceed.

The model was interrogated to determine the following values after taxation:

- Headline values:
  - Net cash flow, NPV at 8% discount rate (NPV_8), IRR
  - Breakeven (NPV_8) nickel price
  - C1 cost /t of Ni (Brook Hunt Methodology) and production year payback.
- Key performance indicators (KPIs):
  - Opex/t nickel, total costs/t nickel, and production payback years.
- Sulphate premium:
  - An assumed US$2,000/t premium for battery grade sulphate in addition to the base nickel revenue.
- Kieserite:
  - As per the market study, the by-product, kieserite, is assumed to yield a net revenue of US$100/t of kieserite produced. This is based on the marketing study (Section 19) which indicated that it could be sold in the Pará state for at least US$180/t. The Steinweg logistics study (Section 18.2) has indicated that freight throughout the state would likely be less than US$70/t which allows US$10/t for handling and bagging.

22.3 Economic model basis

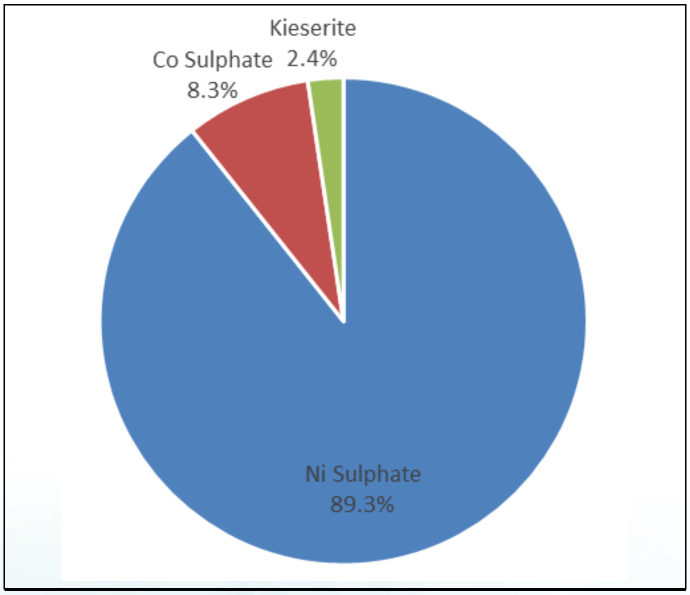
Table 22-3 shows key drivers of the model for the Base Case scenario.

Table 22-3 Model key drivers

Item	Unit	Value
Pre-production period	months	30
Life of Project production	years	38
LOM ROM ore mined and processed	kt	141,316
LOM waste mined	kbcm	12,056
LOM ROM nickel feed grade	%	0.91
LOM ROM cobalt feed grade	%	0.05
LOM beneficiation mass pull	%	51.30
LOM HPAL feed	kt	72,495
LOM beneficiation nickel upgrade factor	factor	152.11
LOM beneficiation cobalt upgrade factor	factor	136.81
LOM average HPAL nickel grade	%	1.39
LOM average HPAL cobalt grade	%	0.07
LOM average HPAL nickel recovery	%	92.00
LOM average HPAL cobalt recovery	%	90.00
LOM average overall nickel recovery	%	71.79
LOM average overall cobalt recovery	%	63.17
Plant HPAL throughput	kt/a	2,000
LOM nickel price (base case)	US$/t	16,400
LOM nickel sulphate premium	US$/t	2,000
LOM cobalt price	US$/t	34,000
Kieserite net revenue	US$/t	100

The proportion of revenue for the LOM for each product sold is shown in Figure 22-1.

Figure 22-1 LOM revenue proportion by product sold



Source: HZM, 2019

22.4 Economic model results

The model results are shown in Table 22-4 and Table 22-5.

Table 22-4 Project economic performance (post-tax)

Item	Unit	Nickel price basis (US$/t Ni) *	
		Base Case (consensus) 16,400	WM Long Term 19,800
Net cash flow	US$ M	7,304	9,546
NPV_8	US$ M	1,722	2,373
IRR	%	26.3%	31.53%
Breakeven (NPV_8) nickel price	US$/t	7,480	7,480
C1 cost (Brook Hunt)	US$/t Ni	8,029	8,029
C1 cost (Brook Hunt) years 1–10	US$/t Ni	7,483	7,483
Production year payback	years	4.2	3.6
LOM nickel recovered	kt	924.0	924.0
LOM cobalt recovered	kt	46.61	46.61
LOM kieserite produced	kt	4,482	4,482
LOM total revenue	US$ M	19,034	22,175
LOM total costs	US$ M	11,729	12,629
Operating cash flow	US$ M	8,451	10,693
Capital intensity – initial capex/t Ni	US$/t Ni	635	635

**Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t of the by-product, kieserite.*

Table 22-5 Project economic performance (pre-tax)

Item	Unit	Nickel price basis (US$/t Ni) *	
		Base Case (consensus) 16,400	WM Long Term 19,800
Net cash flow	US$ M	10,379	13,509
NPV_8	US$ M	2,342	3,185
IRR	%	28.8%	34.5%
Breakeven (NPV_8) nickel price	US$/t Ni	6,946	6,946
C1 cost (Brook Hunt)	US$/t Ni	8,029	8,029
Production year payback	years	4.0	3.5
Total costs^	US$ M	7,508	7,520
Operating cash flow	US$ M	11,526	14,655

**Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t for the by-product kieserite.*

^ Includes royalties, excludes taxation and capital.

22.5 Production and cash flow summary

The Project physicals (by period) are shown in Table 22-6. They are based on the mining schedule developed by Snowden, and the process plant performance predicted by Simulus.

Table 22-6 Production physicals by period

Project period	1–4	5–9	10–19	Remainder	Total
Ore mined (kt)	32,070	52,347	56,899	-	141,316
Waste mined (kbcm)	2,728	4,835	4,492	-	12,056
Ore processed (kt)	9,364	19,686	41,296	70,970	141,316
Process feed nickel grade (%)	1.20	1.16	0.95	0.78	0.91
Beneficiated ore to HPAL (kt)	4,334	10,000	20,000	38,161	72,495
Beneficiated nickel grade*	2.06	1.78	1.54	1.10	1.39
Nickel in product (kt)	81.95	163.77	282.47	395.76	923.96
Cobalt in product (kt)	4.41	8.74	15.84	17.62	46.61

The after-tax cash flow generated in the model (by period), is based on the physicals shown in Table 22-7.

Table 22-7 Project financials by period – post-tax

Project period	1–4	5–9	10–19	Remainder	Total
Revenue (US$ M)*	1,696	3,395	5,881	8,061	19,034
Operating costs (US$ M)	655	1,502	3,389	5,037	10,583
Capital expenditure (US$ M)	1,104	4.60	6.27	31.64	1,146
Net cash flow (US$ M)	(62)	1,888	2,486	2,992	7,304
Pro-rata cash cost (US$/t Ni)	8,058	8,206	8,691	9,442	8,126
Pro-rata cash cost (US$/lb Ni)	3.66	3.72	3.94	4.28	3.69

**Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t for the by-product kieserite.*

22.6 Key performance indicators

The Project LOM KPIs after taxation are presented in Table 22-8.

Table 22-8 Project KPIs after taxation

Item	Unit	Nickel price basis (US$/t Ni)	
		Base Case (consensus) 16,400	WM Long Term 19,800
Value of product sold	US$/t ore	134.69	156.92
Cash cost	US$/t ore	2,680	2,684
Total cost	US$/t ore	83,000	89,365
Production year payback	year	4.2	3.6
Pro-rata cash cost*	US$/lb Ni	3.69	3.69
Pro-rata cash cost	US$/t Ni	8,126	8,139
Pro-rata total cost**	US$/lb Ni	5.76	6.20
Pro-rata total cost	US$/t Ni	12,695	13,668
Brook Hunt methodology C1 cost^	US$/lb Ni	3.31	3.31
Brook Hunt methodology C1 cost	US$/t Ni	7,286	7,286

**The pro-rata cash costs include all direct opex plus royalties.*
***The pro-rata total costs include pro-rata cash costs plus capital costs and taxation.*
^The Brook-Hunt methodology C1 costs include all direct operating expenses excluding royalties.

22.7 Sensitivity analysis

The model was used to prepare a sensitivity analysis for the NPV_8 for the Project after taxation. The sensitivity analysis was completed on the following variables:

- Grade, recovery and price of nickel
- Grade, recovery and price of cobalt
- Net revenue from kieserite
- Pre-production and production capital
- Train 2 capital
- Foreign exchange rate
- Sulphur price
- Power price
- Discount rate
- Overhead costs.
- Beneficiation efficacy.

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 22-9.

Table 22-9 Sensitivity table for the Base Case (US$16,400/t*) NPV_8, after taxation

Sensitivity parameter	-30%	-20%	-10%	0%	10%	20%	30%
Price/Grade/Recovery of nickel	661	1,016	1,369	1,722	2,074	2,427	2,779
Price/Grade/Recovery of cobalt	1,617	1,652	1,687	1,722	1,757	1,792	1,827
Net revenue from kieserite	1,693	1,703	1,712	1,722	1,731	1,741	1,751
Pre-production capital	1,873	1,823	1,772	1,722	1,671	1,621	1,570
Stage 2 capital	1,802	1,775	1,749	1,722	1,695	1,668	1,642
Mining cost	1,799	1,773	1,748	1,722	1,696	1,670	1,645
FX rate	1,535	1,613	1,674	1,722	1,761	1,794	1,821
Sulphur price	1,911	1,848	1,785	1,722	1,659	1,596	1,532
Power cost	1,735	1,730	1,726	1,722	1,718	1,713	1,709
Discount rate	2,523	2,217	1,952	1,722	1,521	1,345	1,189
Beneficiation efficacy	1,298	1,439	1,581	1,722	1,863	2,004	2,146

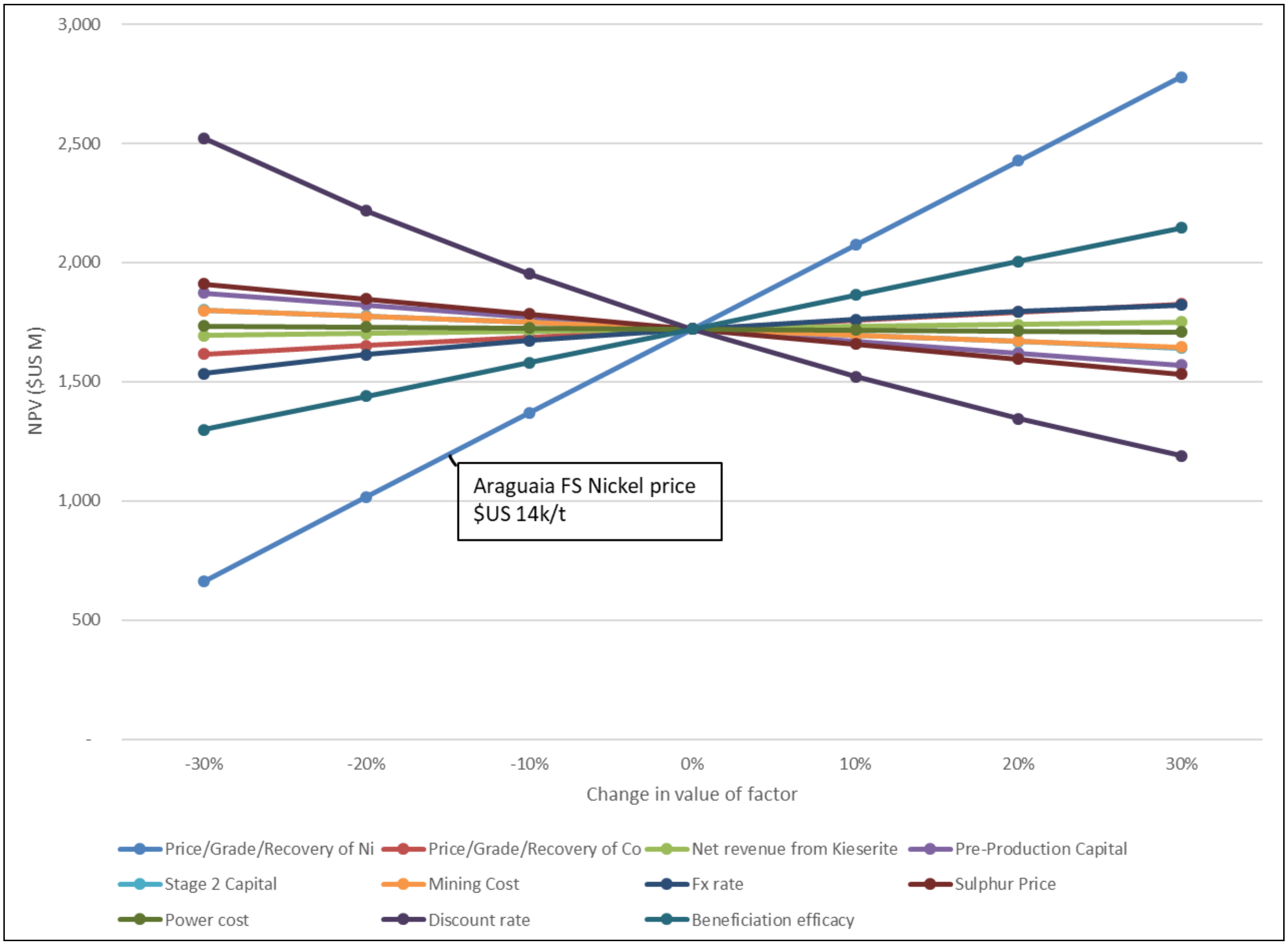
**Includes a US$2,000/t premium for battery sulphate production, US$34,000/t for the cobalt produced as cobalt sulphate, and a net revenue of US$100/t for the by-product kieserite.*

The sensitivity chart is shown in Figure 22-2 and it covers a range of variable changes from -20% to +20%. The flat line in the recovery of nickel indicates that the recovery cannot exceed 100%. As the variable line increases towards the vertical it indicates the Project is more sensitive to changes of that variable.

Figure 22-2 shows that the Project's NPV_8 is most sensitive to nickel price, nickel recovery and nickel grade. This is expected as these three factors directly affect revenue. The Base Case nickel price for the Project is US$16,400/t Ni. The nickel feed grade is well supported by geology and mining engineering studies undertaken. The nickel recovery in the plant is well supported by metallurgical testwork and detailed plant design completed.

The next most sensitive factor is the discount rate applied to the Project which. The discount rate selected (8%) is consistent with other mining projects in Brazil. The third most sensitive factor is beneficiation efficacy.

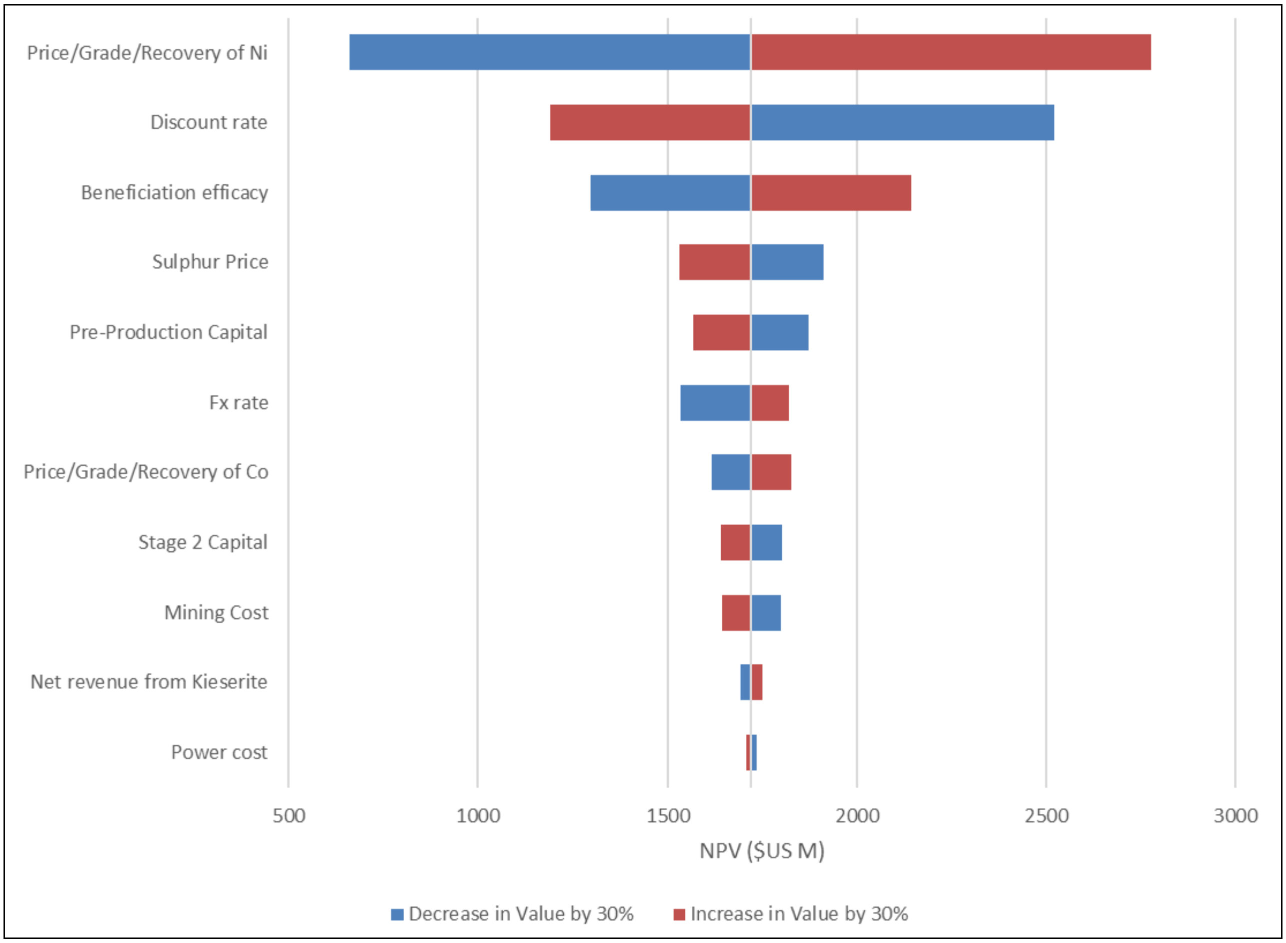
Figure 22-2 Sensitivity to NPV_8 for changes in various key inputs



Source: HZM, 2019

Figure 22-3 is a Tornado plot of sensitivity which shows that the Project's NPV_8 is relatively insensitive to the other factors considered with it being least sensitive to kieserite revenue and power costs.

Figure 22-3 Tornado plot of Project sensitivity



Source: HZM, 2019

22.8 Breakeven analysis

An after taxation breakeven analysis was undertaken for both NPV_8 and net cash flow. This analysis is conducted on the sensitivity analysis data and provides the nickel price which will bring either the NPV_8 or net cash flow to US$0.00. The results of this analysis are presented in Table 22-10.

Table 22-10 Breakeven analysis after taxation

Item	Unit	Breakeven price
Net cash flow	US$/t Ni	5,122
NPV_8	US$/t Ni	7,483

23 ADJACENT PROPERTIES

There is no information from adjacent properties applicable to the Project for disclosure in this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Implementation plan

24.1.1 Introduction

A preliminary Project Implementation Plan has been developed to provide an overview of how the Project may be implemented. A more detailed Project Implementation Plan will be prepared during the Definitive Feasibility Study (DFS) phase of the Project, with the involvement of all disciplines, functions and departments. This document will be updated several times during the Project lifecycle.

Key to the success of the overall Project will be to:

- Clearly identify Project drivers and key success factors
- Selection of appropriate contract delivery strategy
- Alignment of objectives between HZM and key contractors
- Recruitment and assignment of experienced, competent human resources to the Project, committed to working in a spirit of cooperation, trust and openness
- Retain flexibility in DFS for funding early works that will allow accelerated Project start-up
- Employ effective and functional Project Management and Controls during project delivery.

24.1.2 Strategy

In this study, an EPCM contracting strategy has been assumed. It is proposed to have a high-quality owners team embedded within the EPCM team. This will be broken down into separate WBS. Separate areas will have different contracting strategies assigned to ensure best delivery of the Project. It is noted that the Project is at an early stage of development and that it is likely the contracting and implementation strategy will be updated and optimised as the Project develops further. The final selection of delivery strategy for the Project is a complex process requiring significant input from HZM. It will be finalised during the DFS stage when additional information is available that will enable the most effective solution to be selected.

24.1.3 EPCM methodology

The EPCM Contractor will assume responsibility for direct management of all engineering, procurement and construction activities on the Project acting as Owners Agent with all contracts and purchases placed directly by the Owner.

The contractors will be responsible for managing major aspects of the development, while HZM retains overall accountability for the development with specific responsibilities for:

- Project funding
- Overall project governance
- Regulatory approvals
- Third party infrastructure interface management
- Regulatory submissions
- Geological exploration, mapping, resource assessment.

HZM, with the assistance of the EPCM Contractor, will develop the governance structure to support Project delivery.

The scope of services for the EPCM Contractor typically would include:

- Management and supervision
- Engineering and technical services
- Procurement and contracting services, including inspection services, logistics and on-site material control
- Construction management and supervision services, including management of all contractors
- Establishment of an integrated project management planning/forecasting system to monitor, forecast, and report progress and value of work undertaken covering home office, fabrication and construction
- Implementation and maintenance of a QA system, and complying with all QA requirements
- Implementation and maintenance of health, safety and environment systems and complying with all health, safety and environment requirements
- Providing all support and data requested by HZM in obtaining those consents, approvals, licences, etc. that are required for the installation and operation of the Project facilities
- Application of project lessons learned
- Commissioning and handover of facilities
- Close-out and handover of documentation and knowledge to HZM at Project completion.

24.1.4 Management structure

HZM Owners team will be responsible for the management of the main EPCM Contractor and any minor contractors and consultants not directly managed by the EPCM Contractor. The Owners team will be embedded within the EPCM team.

24.1.5 Procurement strategy

The procurement strategy for equipment is to use reputable global vendors but with a local presence. This enables the use of local fabrication as much as possible to minimise local import duties.

Both the cities of Parauapebas and Marabá are near the proposed process plant. These cities already have a substantial amount of infrastructure to enable them to supply services to the local Carajás iron ore mine. Multidiscipline skids and modules could be fabricated in prequalified workshops at these cities and then transported to site for immediate installation.

24.1.6 Implementation schedule

A Project implementation schedule has been developed for the design and construction of the Project. The schedule relies on the procuring of the long-lead items soon after commencement of the works and completing the design and detailed drafting to allow the fabrication and procurement of less critical components. The schedule is based on 33 months from approval to proceed until practical completion. Commissioning is scheduled for a further three months.

Table 24-1 shows the timing planned for key activities in the schedule.

Table 24-1 Key activities in the schedule

Milestone	Schedule
Conditional Letter of Award and full commitment	Week 1
Commencement of process facilities design works	Week 1
Finalisation of plant layout and flowsheets	Week 13
Commencement of major procurement items	Week 13
Commencement of site earthworks and infrastructure	Week 35
Commencement of procurement of skid equipment items	Week 38
Commencement of offsite skid fabrication works	Week 46
Commencement of acid plant construction	Week 54
Commencement of civil and structural works	Week 56
Commencement of site based mechanical installations	Week 88
Commencement of site-based skid installations	Week 96
Commencement of site electrical, instrumentation installation	Week 108
Commencement of site piping installations	Week 108
Commencement of commissioning	Week 138
Practical completion	Week 158

The schedule presented is considered conservative and achievable. It is not a best-case schedule. The major drivers are:

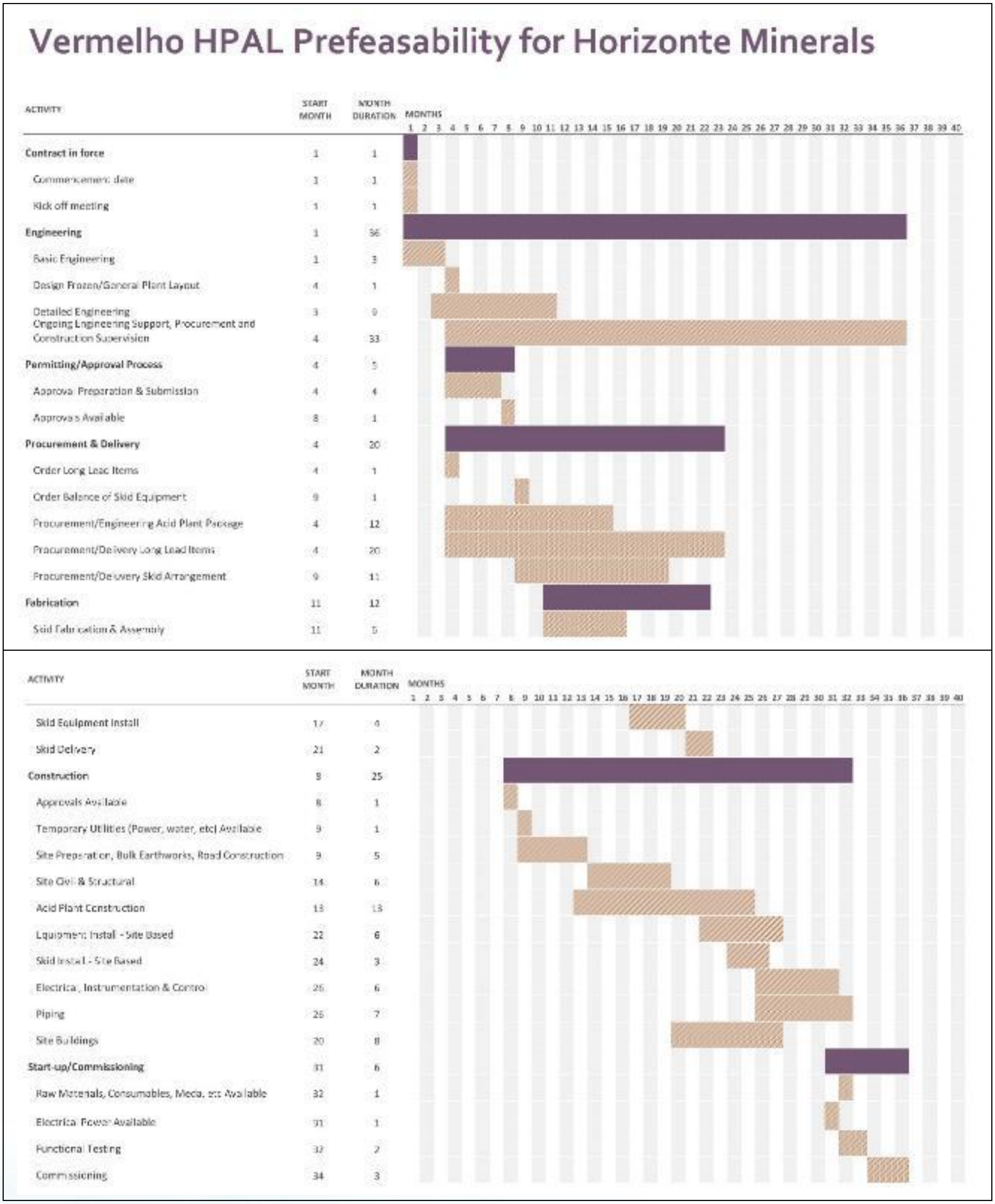
- Long-lead equipment supply items, including:
  - Sulphuric acid plant package – 95 weeks
  - HPAL autoclave – 84 weeks
  - Crystallisation packages – 80 weeks
  - Flash vessels – 60 weeks
  - Autoclave feed pumps – 52 weeks
  - Filter packages – 48 weeks
  - Skid fabrication and fit out – 44 weeks.
- Consecutive site-based activities:
  - Earthworks and roads – 21 weeks
  - Concrete and site structural steel – 30 weeks
  - Site based mechanical equipment – 26 weeks
  - Skid installation – 12 weeks
  - Site electrical, instrumentation and control – 26 weeks
  - Site piping – 30 weeks.

As the design proceeds and equipment orders are placed, more certainty around equipment lead time and scheduling of the skid fabrication priorities may enable a reduction in the proposed schedule. Areas that may enable an improved schedule are:

- Expedite or staggered delivery of key structural and equipment skids – possible eight-week gain
- Site works tenders and contracts accelerated ahead of completion of engineering– possible four-week gain
- Overlap site concrete and structural activities with completion of site preparation as progress is completed in defined areas – possible four-week gain.

A Gantt chart of the proposed implementation schedule is shown in Figure 24-1 below.

**Figure 24-1	Implementation Gantt chart**



Source: Simulus, 2019

24.1.7 Construction expenditure

Where equipment lead times for major packages are shorter than the critical path items, placement of equipment orders are to be scheduled as late as possible to meet installation timing. The payment structure is based on an initial deposit, three progress payments and a retained amount for all equipment and materials packages.

24.1.8 Operational Readiness

Operational Readiness systems intend to accomplish a successful commissioning, start-up and ramp-up process and achieve sustainable operations, with assets that are safely delivered on-time and on-budget, achieving the operational expectation. Operational Readiness frameworks and asset management systems through all phases of the Project lifecycle attain faster, safer, more reliable start-ups, which allows for quicker transitioning into ramp-up and ongoing improvements and performance. This is undertaken through:

- Initial operational readiness definition, risk identification, strategic planning
- The right systems including asset management, maintenance planning, procedures, methods, plans, overarching management systems, etc.
- The right people including recruitment, adequate resourcing, training, etc.
- The right equipment, i.e. fit for purpose, properly sized and maintained, of appropriate quality and sufficient redundancy with critical spares and stock management
- Meeting statutory commitments and community expectations
- Functional readiness and alignment across all departments and disciplines (e.g. operations, engineering and maintenance)
- Sustainable operations with a focus on continuous improvement.

The PFS phase of study does not require detailed Operational Readiness frameworks, Project delivery systems or plans. These will be initiated during the DFS phase of study.

24.2 Other relevant information

There is no other relevant data and information to disclose.

25 INTERPRETATION AND CONCLUSIONS

The Vermelho project by HZM is within the Carajás Mining District, an active mining region with well-developed infrastructure, skilled workforce and services to support development of the Project. The previous operator had completed a FS and a previous production decision to produce nickel and cobalt from open pit mining of the V1 and V2 deposits and processing via the HPAL method.

In October 2018, HZM completed an FS on its Araguaia ferro-nickel project located between 85 km and 150 km to the south of the Project. Consequently, HZM is familiar with working in the region and the associated costs and consequently is well placed to consider synergies that may exist in the development of the Araguaia and Vermelho projects.

25.1 Mineral Resources

The Vermelho area was explored in various stages by Vale from 1974 to 2004 involving approximately 152,000 m of combined drilling and pitting (Table 25-1).

The drilling grid density was substantially enhanced in 2002 to 2004, and most of the resources were upgraded to the Measured category as defined by CIM Definition Standards. Pilot plant metallurgical studies were conducted in Australia. A PFS was prepared in 2003, and a FS was completed in August 2004 by GRD-Minproc (2005). This study confirmed the positive economic outcomes obtained in previous studies and showed production capacity of 46,000 t/a of metallic nickel and 2,500 t/a of metallic cobalt. Vale elected to place Vermelho on hold after delivery of the FS due to liquidity issues.

Table 25-1 Summary of Vale exploration

Description	V1		V2		Project area		Total	
	Unit	Drilling (m)	Unit	Drilling (m)	Unit	Drilling (m)	Unit	Drilling (m)
Pit excavation	251	3,903	231	3,585			482	7,487
Large-diameter drilling	86	3,582	152	7,333			238	10,915
Diamond drilling	39	2,931	24	1,806			63	4,736
Rotary percussion	1,289	72,875	867	50,768	65	1,370	2,221	125,013
Exploration pits					11	60	11	60
Mixed geotechnical drilling					20	788	20	788
Percussive geotechnical drilling					75	783	75	783
Auger drilling	15	8	13	8	296	265	324	280
Rotary geotechnical drilling	9	945	9	906			18	1,851
Totals	**1,689**	**84,243**	**1,296**	**64,405**	**467**	**3,265**	**3,452**	**151,913**

Source: Extracted from Vale, 2004

The geological setting of the Vermelho nickel-cobalt deposits is well understood. The deposits consist of two hills named V1 and V2 aligned on a northeast-southwest trend, overlying ultramafic bodies. The ultramafic bodies have had an extensive history of tropical weathering, which has produced a thick profile of nickel-enriched lateritic limonite and saprolite.

The V1 and V2 deposits form flat lying topographical highs. The V1 hill extends for approximately 2.4 km east-west, ranging from 700 m to 1.6 km north-south. The V2 hill has an east-west elongation, and extends for approximately 1.9 km east-west, ranging from 600 m to 900 m north-south. The V1 deposit has an average thickness of 53 m and a maximum thickness of 146 m, whereas the V2 deposit has an average thickness of 56 m and a maximum thickness of 115 m.

Independent reviews of Vale's operations at Vermelho have found that survey, drilling, sampling procedures and sample preparation and analysis protocols were adequate to support the estimation of Mineral Resources. During February 2018, the author reviewed the historical MREs (Section 6.2) and concluded that the CVRD_25_2_m MRE that was audited by Snowden in 2005, is appropriate for adoption as a current MRE and concluded that the reporting conforms to the requirements of the CIM Definition Standards (2014).

Within the mining licence, at a cut-off grade of 0.7% Ni, a total of 140.8 Mt at a grade of 1.05% Ni and 0.05% Co is defined as a Measured Mineral Resource; and a total of 5.0 Mt at a grade of 0.99% Ni and 0.06% Co is defined as an Indicated Mineral Resource. This gives a combined tonnage of 145.7 Mt at a grade of 1.05% Ni and 0.05% Co for Measured and Indicated Mineral Resources. A further 3.1 Mt at a grade of 0.96% Ni and 0.04% Co is defined as an Inferred Mineral Resource at a cut-off grade of 0.7% Ni.

25.2 Metallurgical testing and process design

The Project has been extensively studied and tested from a metallurgical point of view. Finished metal products (LME grade nickel cathode) were produced with high metal recovery by the former project owners Vale. The current Project reflects a change in strategy by the current project owners, HZM, to produce high purity nickel and cobalt sulphate products suitable for the battery market, which is expected to achieve a premium to the nickel and cobalt metal price.

Based on the previous testwork completed by Vale (2002–2004) and new testwork completed by HZM (2018–2019), Simulus has selected the beneficiation, HPAL, MSP, POX, CoSX and crystallisation as the preferred process flowsheet. The selected flowsheet via MSP intermediate product is proven at an industrial scale and provides a more selective (less impurities) process, facilitating the production of high-purity nickel and cobalt sulphate. The front-end design remains consistent, the "back end" of the plant changes. The capital cost is expected to be lower than the traditional hydrogen reduction processing route post MSP and the premium from the production of metal sulphates justifies the change in the process selection.

A major positive aspect of the Project is that the ore is amenable to upgrading by removal of a significant mass proportion of free silica. Apart from increasing the grade of nickel and cobalt, this has the beneficial effect of reducing the amount of material that has to be leached in the HPA.

Extensive beneficiation and HPAL testwork completed by the former owners are directly applicable to the current Project, including 550 batch variability beneficiation tests, 100 batch variability HPAL tests and multiple campaigns of pilot testwork during prefeasibility and final feasibility phases. Samples used were representative of the LOM and were appropriately geographically spread, over depth, by ore types and lithologies.

Nickel recovery and upgrade from previous beneficiation testwork were used in the current design, resulting in a proposed peak feed rate of ROM ore of 4.34 Mt/a at an average grade of 1.07% Ni which is upgraded to 2 Mt/a at an average grade of 1.85%. Similarly, HPAL metal extractions and acid consumption from previous Vale testwork is directly applicable to the current Project and has been used as the basis of design. Metal extraction is 97% for both nickel and cobalt at a 347 kg/t acid dose.

MSP was produced during the PFS pilot testwork completed by Vale in 2002 with a grade of 54.6%, 3.22% and 37.3% for nickel, cobalt and sulphur respectively. Impurities were 0.4%, 3.6%, 0.24% and 40 ppm for aluminium, iron, silicon and manganese respectively and is suitable for feed to POX and subsequent CoSX and crystallisation to produce high-purity nickel and cobalt sulphate.

The downstream processes of POX, CoSX and crystallisation have not yet been demonstrated at pilot scale. However, these are conventional and proven industrially processes and appropriate industry benchmark data and industry experience has been used in the Project plant design. An integrated pilot test campaign with the selected flowsheet of beneficiation, HPAL, MSP, POX, CoSX and crystallisation is recommended for the next stage of project development. The process design nickel and cobalt recovery from HPAL feed to sulphate crystal product is 94.4% and 94.9% of the nickel and cobalt, respectively.

The current design has applied a zero liquid discharge and dry stacking of leach residue solids to minimise environmental and safety concerns. As a result, a kieserite (magnesium sulphate monohydrate) by-product will also be produced. The kieserite product quality requires confirmation during pilot testing but is expected to be suitable for sale to the fertiliser market (Brazil and international).

The HZM testwork completed in 2019 and described in Section 13.5 demonstrates that it is possible to produce nickel and cobalt sulphate from Vermelho ore, the latter having been tested to very near the purity level required for the battery market. It is recommended that full pilot-scale testing is undertaken on a bulk sample composite that replicates the final version of the flowsheet proposed.

25.3 Mineral Reserves

This PFS demonstrates that the Mineral Resources can be technically and economically extracted in a jurisdiction which has a long history of successful mining. The 145.7 Mt of Measured and Indicated Mineral Resource at 0.7% Ni cut-off has been converted to 141.3 Mt of Probable Mineral Resource.

25.4 Project economics

The study has developed the costs for the Project to a PFS (±25%) standard and given the assumptions demonstrates the positive economics of the Project with a Base Case post-tax NPV_8 of over US$1.7 billion with a payback period of under five years.

25.5 Infrastructure and logistics

The Project is located in a mining district so is well served by local infrastructure. The study has developed designs and costs for the additional infrastructure required to develop the Project. This includes an 11 km access road, a 15 km powerline, a 5 km pipeline, a mine residue storage facility and a mine sediment control dam. The cost or technical challenges presented by this infrastructure is well understood and moderate.

25.6 Social, environment and permitting

The current Project status is summarised as:

- The previous EIA RIMA, LP and all environmental licences relating to Vermelho mine and process plant contain background studies are useful to guide new social and environmental studies in the region
- Once the concept design for the project is finalised by the technical team within HZM, a new social and environmental impact assessment (EIA RIMA) will be conducted with the objective of obtaining an LP, and later LI to construct the Project in line with HZM’s proposed Project characteristics
- HZM will require engagement with community groups in the region, as well as authorities, including:
  - SEMAS – Pará State Environmental Licensing agency
  - INCRA – Brazilian Land Authority
  - ITERPA – Pará State Land Authority
  - IPHAN – Brazilian Institute of Historic and Artistic Heritage

– Municipal level authorisations.

The Project is located in a region where mining has promoted significant transition over recent years. Economic growth has led to advancements such as, new infrastructure, thriving local business, increased urban population, a skilled local workforce, an increased consumer confidence, and local authorities familiar with the needs and challenges of mining.

Whilst new baseline studies will be undertaken on the physical and biological environment in the area, HZM estimate that the impacts will be limited as the Project is located on a large farm which has already seen deforesting activities take place, although vegetation clearance will be required to mine the areas.

The main impacts anticipated are landscape changes, fauna displacement, atmospheric pollution, resettlement, pressure on water availability and liquid effluent generation from dry-stack tailings and waste-rock dumps. Further baseline studies will inform environment control plans; however, given the previous approval of a much larger HPAL process by Vale more than 10 years ago, HZM's team anticipates this will be assessed as a viable project form an environmental and social aspect.

The previous studies demonstrated no intersection of indigenous or Quilombola[10] communities.

With respect to vegetation and legally protected areas, the Project limits are outside of any National Forests or Conservation Units.

25.7 Risks

25.7.1 Mining risks

Key mining risks include:

- Trafficability, wet weather and the related issue of selecting the mining equipment fleet
- Ore control and dilution with sub-grade material
- The failure to reach production targets as a consequence of grade control activities or unexpected occurrence of gabbro dykes, pinnacles or boulders
- Predictability of grade when reclaiming from large long-term stockpiles
- Mining contractor costs being materially higher to those predicted in this study.

There are several mitigating factors related to trafficability and wet weather:

- The wet season is confined to six months of the year
- Based on the silica content of the orebody, the first six to seven years of the mine life will be undertaken in areas of generally good trafficability. This will provide an opportunity for the operation to learn to deal with this issue in a relatively easier environment.

Ore control challenges will be met by:

- Implementation and ongoing refinement of the mine-to-mill strategy which includes 12.5 x 12.5 grade control drilling well ahead of mining
- Several sample points along the or supply chain to the plant and in the plant to control and deal with anomalies
- Large stockpiles of various categories of ore so that the blend can be met on a daily basis from stockpiles
- The ability to direct tip from the mine.

[10] Settlements first established by escaped slaves

Various ways of managing the risk of running out of ore supply due to wet weather have been suggested in this report. They include:

- Maintaining up to three months' ore supply on the ROM pad
- Using all-wheel drive trucks for 20% of the material movement
- Draining water from active mining areas
- Keeping haul roads in good condition (through design and maintenance)
- Sheeting haul roads and active working areas with beneficiation tailings or ferricrete
- Compacting or rolling long-term dump surfaces to minimise water ingress.

In addition to these suggestions, it will be possible to assess the suitability of the mine equipment fleet through the life of the mine and replace equipment to suit the conditions. While it is not desirable for the entire truck fleet to be made up of all-wheel drive trucks, this option has been costed and shown to be a minimal part of the Project cost.

Stockpile management will be an important part of the Vermelho operation. While there is a risk that it will be difficult to predict grade when feeding ore from long-term stockpiles, there are a number of mitigating factors, including:

- Large volume reclaim from stockpile will only commence in year 18 of the Project
- The division of the stockpiles into Ni and Mg grade ranges, will provide the option of using material from different stockpiles to manage unexpected grades.

Other procedures that should be put in place to minimise this risk include:

- A comprehensive stockpile management system should be implemented. This system will gather data from the truck dispatch system as the primary input. Stockpile dumps should be designed and divided into levels and blocks (containing approximately a week's ore supply) with unique names. A database should keep track of the grade of material mined and where this material is stockpiled.
- Stockpiles should be protected against erosion by vegetating the dump faces.
- Stockpiles should be protected against water ingress by rolling the top surfaces and by building drains to lead water away from the dump.

25.7.2 Metallurgical testing and process design

Overall, the metallurgical risks associated with the Project are considered to be low. With the exception of the final crystallisation step for nickel and cobalt, the remaining process and technology being applied are standard, industry proven, robust and conventional processes. This includes HPAL, MSP, POX and CoSX as the main process steps. The crystallisation of nickel and cobalt sulphate salts is itself an industrially proven process, but not in the context of producing battery-grade materials directly at the mine site from a nickel laterite project.

A review of the proposed process plant design, considering the selected flowsheet and products being produced and the current state of metallurgical testwork data available, identified the following process risks:

- Actual nickel and cobalt grades in the HPAL feed is less than expected due to lower than expected upgrade in the beneficiation area. This risk was considered low due to the extensive testwork that has been completed to date which includes 550 variability tests on 13 t of samples derived from 6" core which were considered representative of the LOM, ore types and lithologies. The upgrade model and equations developed were validated during 160 t of pilot campaigns which included 12 composites comprised of over 6,000 samples. The equations and supporting the data was independently reviewed by Geostats (Geostats, 2004). Despite the exhaustive testing HZM has elected to reduce the predicted upgrade by 25% for all ore blocks in the financial evaluation to further mitigate this risk.

- Higher than expected acid consumption in the HPAL process. This risk was considered low, due to the extensive testwork that has been completed to date including 100 batch variability HPAL tests, three continuous week-long pilot campaigns during Vale PFS testwork and four continuous two-week long pilot campaigns during Vale FFS testwork which gave predictable acid consumption based on upgraded ore feed composition.
- Ability to achieve nickel and cobalt product purity required for sale into the battery market. This risk was considered low to moderate due to the status of current testwork which has not yet demonstrated the level of purity required. However, the process flowsheet adopted is considered suitable for the production of high-purity products and the risk may be ameliorated with further testwork in future stages of the project development. Additional impurity removal processes and technology is available to augment the existing process flowsheet if the need is identified in future testwork.
- Marketing and sale of the kieserite crystal by-product. The specification of this by-product has not yet been demonstrated via testwork but is expected to be suitable for the fertiliser industry. This is an industrially proven process, so the technical risk is associated with the composition of the feed stream to the kieserite crystalliser and kieserite product composition produced. Some disposal cost may be incurred for off-specification kieserite if produced. Given that this was a low value product that was not contributing significantly to the Project economics, the risk is considered low.
- Inability to consistently achieve the zero-liquid discharge (i.e. have a net zero water balance for the site) that leads to the requirement for an effluent disposal stream which would likely require some treatment (likely to be lime precipitation and pH adjustment) environmental permitting and associated cost and delay. This risk is considered to be low to moderate, given the engineering design completed to date. Possible risk mitigation includes increase the design margin for the kieserite crystalliser system, diversion of excess steam from the acid plant (that is currently used to produce electricity) to evaporate more water and/or installing additional surge capacity for barren effluent to even out any short-term fluctuations in processing capacity.
- Poor stability of the HPAL residue solids from a geotechnical point of view that prevents dry stacking of tailings. This risk is considered low to moderate given there are other operations around the world with similar solid leach residue streams that are using this technology, and on the assumption that further work will be completed as required. However, there is no sample available yet to confirm the physical and geotechnical properties to design the dry stack and as such it presents a risk. HPAL residue samples will need to be generated in future testwork campaigns and provided to specialist geotechnical consultants experienced in dry stacking techniques to collect the necessary data to develop an appropriate dry stack design. Possible risk mitigation options include alternative process equipment (filter) selection to improve the physical and geotechnical properties of the wet filter cake and/or blending with the coarser beneficiation rejects to form a more stable residue.
- Overall nickel and cobalt recoveries lower than expected. This risk is considered low due to the engineering and testwork completed to date and the process flowsheet selected which relies on industry proven technologies. Extensive testwork on the HPAL process for this Project has consistently achieved 96% or better metal extraction. High metal recovery is typical for the MSP and POX unit processes which, coupled with recycle of minor refinery bleed streams should provide high overall metal recovery, provided good operating practices are adopted. Apart from the nickel and cobalt that was not extracted in the HPAL process, the main potential loss is of soluble nickel and cobalt from insufficient washing of the HPAL residue solids. This has been accounted for in the current design but should be confirmed via vendor testwork during future pilot testing.

25.7.3 Other risks

Permitting and social

The permitting pathway for the Project is well defined and well established, with several mines in the area in operation permitting is not considered a high risk; however, there is a risk that the Project permitting process could be delayed due to competing priorities within agencies. This is being mitigated by engagement of regulators as HZM move the Project forward. Monthly reporting on permits and notifications should be continued.

If advanced, this would be the first HPAL nickel project to be licensed through to construction in Brazil and so HZM's team will need to support agencies by providing technical/expert advice and assist with benchmarking other HPAL projects around the world. This also means that attracting and retaining appropriately experienced and qualified HPAL operators and technicians is a project risk. This risk will be mitigated be extensive manufacturer training and commissioning programs and rigorous procedure development.

The social licence to operate is considered crucial to the Project's success. The community around the Project in general welcomes mining (as there are other operations in the area). However, there is a risk that community resistance may be encountered, such as a gap between those that work in mining and those that don't (inflation/housing availability); accordingly, HZM will develop and implement plans to engage with the community during development and source local labour and suppliers for products and services where possible during operations.

The rural community living near the Project area must be engaged regularly to ensure any concerns are addressed and ongoing positive relationships are maintained.

Market

The sensitivity study shows that the Project is most sensitive to the nickel price This is a high risk however both the long-term and the spot price of nickel is higher than the Base Case nickel price. Should the sulphate premium be less than expected this will impact negatively on Project economics.

25.8 Opportunities

This PFS demonstrates that the Project is technically and economically robust and should be progressed to FS, detailed design and construction. However, during the course of developing the PFS, a number of opportunities have presented themselves. Some of the higher potential opportunities are discussed below.

25.8.1 Opportunities to increase the value of the PFS Base Case

Process

A review of the proposed process plant design, considering the selected flowsheet and products being produced resulted in the following list of possible opportunities to either reduce cost or improve the process:

- Substitute current 2 x 1,200 t/d Outotec acid plants (Train 1 and Train 2) with a single larger 1 x 2,400 t/d Outotec plant. This will bring forward some capital expenditure but will reduce the overall cost by around ((81 + 78) - 135) = US$24 million in direct costs.
- Substitute current 2 x 1,200 t/d Outotec acid plant and 2 x GE steam turbine generators with cheaper CERE/MECS® brand Chinese supplied plants inclusive of the steam turbine generators.
- Substitute current 2 x 1,200 t/d Outotec acid plant with single 2,400 t/d cheaper Chinese, CERE/MECS® acid plant ((81 + 78) - (111.7)) = US$54.5 million.

- Consider the use of screw presses for the leach residue slurry solid-liquid separation step rather than pressure filters. Potential savings are estimated to be up to 35% of the current costs, or US$26 million.
- Consider the potential use of indirect heating on the autoclave feed and discharge rather than traditional direct contact heater vessels and flash cooling vessels. This technology is used in the alumina industry but is unproved in nickel laterite HPAL plants and is considered to present a higher process risk. It has however been trialled at a demonstration plant scale on an existing laterite operation. Estimated savings of 15% of HPAL area or US$13 million. This may also assist with reduction in maintenance costs.
- Change from closed-circuit cooling water to open-circuit cooling water system. Potential saving (assumed to be 25% reduction of area cost) is only US$3 million.
- Adopt crusher equipment vendor's suggestion that only primary and secondary crushing (roll crusher/mineral sizer) is required rather than the more conservative three stage crushing that has been allowed. Assumed potential savings of US$1 million.
- Depending on the cost of power, consider deletion of the steam turbine generator to reduce capital (at the expense of some operating cost). Potential saving in the order of US$7 million.
- Review consolidation of reagents and utilities between the two trains and consider building a slightly larger version that is common to both trains rather than two smaller units, e.g. process water pond, plant air compressors, instrument air driers, water systems, limestone grinding circuit, flocculant plants etc. Assuming an approximate 30% overall saving but with some capital being brought forward gives potential saving of US$5.7 million.
- Review equipment suppliers to select second-tier vendors, e.g. instead of Wirth or Weir (Geho) autoclave feed pumps, consider a cheaper Chinese alternative, which may save costs at the expense of higher risk. Assume a total saving across the Project direct mechanical equipment costs of US$5 million.
- Investigate options to produce alternative nickel and cobalt battery material product such as ternary precursor which can trade at a premium to the nickel and cobalt content of approximately 140%.
- Investigate options to produce alternative products that are currently being leached but which are disposed of as waste, for example high purity alumina and/or manganese products, either as manganese metal, sulphate or dioxide, thereby providing additional revenue streams for low marginal cost.

Mining

Direct tipping from mine to ROM hopper

It is envisaged that after production commences, the grade control processes put in place prior to mining will sufficient to predict the mined grade and that material may be direct tipped from the mine face directly to the plant feed bin. This will result in significant savings in mining costs by reducing re-handle.

Mining cost reductions

The FS developed mining costs are based on proposals from contract mining operators in Brazil. These proposals were solicited using the mine schedule and with no negotiation as there was no imminent contract. It has been recognised that when HZM is in position to award a large mining contract that negotiations should result in significant mining unit costs savings.

25.8.2 Potential to develop the Project as a ferronickel project rather than an HPAL project

Early studies of Vermelho by Vale focused on smelting the saprolitic portion of VMH ore to ferronickel, however, this was later switched to a study of hydrometallurgical processing of the more limonitic ores (this forms the Base Case of this PFS). Based on laboratory testwork HZM carried out during the last quarter of 2018 and earlier work on HZM's the nearby Serra do Tapa deposit by the former project owners, as well as the close similarity of the chemistry of the blended feed over the first 10 and subsequent years to existing RKEF operations, the RKEF process is also considered appropriate for the treatment of the Vermelho and Serra do Tapa saprolitic material; the ferronickel product would have a nickel content of 30% Ni.

Consequently, Snowden has developed a scoping study case that explores the opportunity that utilises the Mineral Resources of the Project, in addition to HZM's Mineral Resources at the nearby Serra do Tapa deposit, about 120 km by road from Vermelho to feed a 1.8 Mt/a ferronickel plant using the RKEF process route to produce ferronickel.

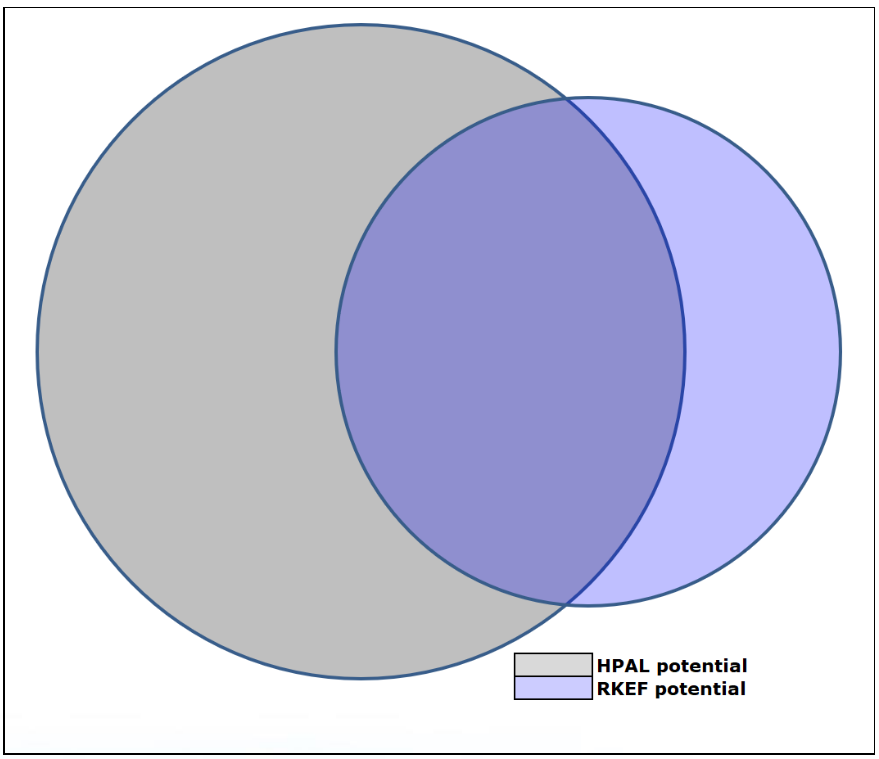
The case that Snowden developed includes the development of mine plan and schedule, process design, process plant design, and detailed capital and operating costs which were used to derive economic KPIs.

Mineral Resources for the RKEF option

Vermelho

The RKEF option includes material from Vermelho Mineral Resource (Section 14) some of which is also suitable to HPAL processing and is included the HPAL process route which forms the Base Case of this study (Figure 25-1).

Figure 25-1 Vermelho Mineral Resource potential process route split



Source: HZM, 2019

Serra do Tapa

The Serra do Tapa Mineral Resource was disclosed by HZM in August 2019.

The grade-tonnage curves for selected cut-offs of the Serra do Tapa Mineral Resource is shown in Table 25-2.

Table 25-2 Serra do Tapa grade-tonnage comparison

Ni % cut-off	Geology model				
	Mass (Mt)	Ni (%)	Fe (%)	SiO_2/MgO	Al_2O_3 (%)
0.5	122.5	0.99	13.7	1.96	3.02
0.6	110.5	1.03	14.0	1.99	3.09
0.7	95.7	1.09	14.4	2.04	3.20
0.8	79.9	1.16	14.9	2.12	3.32
0.9	65.1	1.23	15.4	2.23	3.46
1.0	51.9	1.30	15.9	2.35	3.59
1.1	39.7	1.38	16.4	2.50	3.73
1.2	29.1	1.46	16.9	2.67	3.87
1.3	20.9	1.55	17.3	2.82	3.97

Economic outcomes of the RKEF opportunity

A scoping study was completed for all the mining, processing and infrastructure opex and capex for the RKEF option. From this, a cashflow model was developed. The model's outcomes are presented below in Table 25-3.

Table 25-3 RKEF opportunity post-tax economic KPIs

Item	Unit	Nickel price basis (US$/t Ni)	
		16,400	19,800
Net cash flow	US$ M	5,327	7,657
NPV_8	US$ M	1,375	2,072
IRR	%	29.2%	36.8%
Breakeven (NPV_8) nickel price	US$/t	9,784	9,784
C1 Cost (Brook Hunt)	US$/t Ni	7,450	7,450
C1 Cost (Brook Hunt) years 1–10	US$/t Ni	6,651	6,651
Production year payback	years	4.6	3.9
LOM nickel recovered	kt	814	814
LOM Fe recovered*	kt	2,746	2,746
Total revenue	US$ M	13,355	16,124
Total costs	US$ M	8,846	9,285
Operating cash flow	US$ M	6,146	8,475
Capital intensity – Initial capex/t Ni	US$/t Ni	545	545

**Fe credits are not included in the economics.*

26 RECOMMENDATIONS

26.1 General

The PFS demonstrates that the Project is technically, economically, and should clear all the regulatory and permitting requirements. Consequently, the Project should progress to a FS after completing additional verification drilling, trial mining, metallurgical testing, geotechnical investigation, and further environmental and social field programs.

26.2 Geology

Given the last drilling was completed 15 years ago, a program of twin analysis should be completed. This can likely be completed as part of any future metallurgical testwork program. A limited program of chemical re-analysis should be conducted using best available certified reference standards for audit and verification purposes.

The beneficiation performance is material to the viability of the project. Further work is recommended to refine basis of the beneficiation as it relates to the various mineralogical and lithological domains:

- Logging of free silica within existing core
- Mineralogical testing of variability samples.

The MRE was completed over 15 years ago, and there have been other partial model updates since, which have mainly focused on adjustments to the domaining scheme. The author recommends that the MRE is updated:

- Resolving discrepancies between resource model topography and later, more detailed, topography data
- Critical review and updating (if appropriate) of the domaining and geological modelling of the deposit
- Estimation using appropriate unfolding and estimation techniques
- Produce the block model in a similar format to other HZM deposits to enable ease of reporting, and comparison.

26.3 Mining

The follow is recommended prior to or part of an FS:

- The Project should convert to one topographic datum
- Undertake trial mining and test pitting to evaluate the ability to do bulk extraction versus the requirement for selective mining around gabbro dykes, boulders/pinnacles etc.
- A thorough study into the trafficability of typical mining conditions and how it is likely to be affected by weather conditions. More quantitative tests need to be undertaken to characterise size distribution and material properties of typical rock types. This should include a study of the availability of ferricrete and properties of beneficiation tailings and the potential suitability of these material for sheeting material. In addition to this, detailed plans should be outlined for managing large volume rainfall. Trafficability on ore stockpile and waste dumps should also be considered.
- A study into the management of water associated with the proposed mine plan should be completed and mitigations recommended.
- Mining contractor budget pricing should be completed, and a more comprehensive equipment study completed.
- Assess quantities and the distribution of material amenable for direct tipping into feed hoppers.

- A review of the geotechnical data in the context of the proposed design should be completed with any design improvements suggested.
- A mine-to-mill study should be completed with key stakeholders.
- An overall mine layout review should be completed with key stakeholders to optimise the use of space.

26.4 Metallurgy

The following recommendations relating to metallurgy and process design have been made throughout the report but are summarised here as follows:

- An integrated pilot testwork campaign should be completed in future stages of Project development that reflects the current proposed process flowsheet incorporating beneficiation, HPAL, MSP, POX, CoSX and crystallisation. The intent would be to demonstrate:
  - beneficiation performance
  - nickel and cobalt sulphate and kieserite product quality
  - overall metal recoveries
  - reagent selection and consumptions
  - generates process design data necessary for design, particularly for MSP and downstream processes
  - sample generation for equipment vendor tests and sizing.
- Geotechnical testing of leach residue and dry stack tailings storage design to be finalised.
- Completion of a cost-benefit analysis on process design opportunities presented in Section 25.8.1.
- Complete an updated FS based on the outcome of above recommendations.

26.5 Permitting, social, environment

It is recommended that as HZM will recommence the licensing program for Vermelho, some studies should be updated to reflect the current physical, biological and social settings. The main studies should include:

- Integrated EIA RIMA, including:
  - Water availability and quality
  - Air and noise baseline
  - Soil quality
  - Flora and fauna inventory
  - Socioeconomic diagnosis
  - Resettlement analysis and planning.
- Studies aligning with guidelines set out by IFC Performance Standards and Equator Principles should be undertaken.
- The HZM team should maintain and increase positive working relationships with rural community groups living near the Project area and with local agencies.

27 REFERENCES

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28 CERTIFICATE OF QUALIFIED PERSONS

28.1 Anthony Finch

I, Anthony Finch, Executive Consultant of Snowden Mining Industry Consultants Pty Ltd, Level 6, 130 Stirling Street, Perth, Western Australia do hereby certify that:

- I am a co-author, and the overall qualified person of the technical report titled "NI 43-101 Technical Report on the Vermelho Project, Pará State, Brazil" dated 31 October 2019 (the "Technical Report") prepared for Horizonte Minerals Plc.
- I graduated with a Degree in Bachelor of Engineering, Mining from the University of Queensland in 1986. I am a Member of the Australasian Institute of Mining and Metallurgy. I have worked as a Mining Engineer almost continuously for a total of 43 years since graduation. I have been involved in resource evaluation consulting for 15 years, including reserve estimation of nickel laterite deposits for at least five 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.
- I visited the Vermelho Nickel-Cobalt Project Property from on four occasions between June 2018 and September 2019
- I am the overall Qualified Person for the Technical Report.
- I am responsible for the preparation of Sections 1, 2, 3, 4, 5, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27 of the Technical Report.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the issuer as defined in section 1.5 of the Instrument.
- I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
- 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, Western Australia, on 31 October 2019,

"Original Signed"

Anthony Finch

P Eng, MAusIMM (CP Min)
Executive Consultant
Snowden Mining Industry Consultants

28.2 Andrew F. Ross

I, Andrew F. Ross, Executive Consultant of Snowden Mining Industry Consultants Pty Ltd, Level 6, 130 Stirling Street, Perth, Western Australia do hereby certify that:

- I am a co-author of the technical report titled “NI 43-101 Technical Report on the Vermelho Project, Pará State, Brazil” dated 31 October 2019 (the “Technical Report”) prepared for Horizonte Minerals Plc.
- 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 Mining and Exploration Geology from James Cook University of North Queensland. I am a Fellow of the Australasian Institute of Mining and Metallurgy. I have worked as a geologist almost continuously for a total of 47 years since graduation. I have been involved in resource evaluation consulting for 22 years, including resource estimation of nickel laterite deposits for at least five 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.
- I visited the Vermelho Nickel-Cobalt Project Property from 9 to 12 December 2003 and on 5 August 2019.
- I am responsible for the preparation of sections 6 to 12 and 14 of the Technical Report and parts of section 1 and 26
- I am independent of the issuer as defined in section 1.5 of the Instrument.
- I have no prior involvement with the property that is the subject of the Technical Report apart from the site visit in 2003 during the drilling program conducted by Vale.
- I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
- 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, Western Australia, on 31 October 2019,

“Original Signed”

Andrew F. Ross

M Sc, FAusIMM
Executive Consultant
Snowden Mining Industry Consultants

28.3 Simon Walsh

I, Simon Walsh, Principal Metallurgist of Simulus Pty Ltd, 82 John Street, Welshpool, Western Australia do hereby certify that:

- I am a co-author of the technical report titled "NI 43-101 Technical Report on the Vermelho Project, Pará State, Brazil" dated 31 October 2019 (the "Technical Report") prepared for Horizonte Minerals Plc.
- I am a graduate of Murdoch University (BSc Extractive Metallurgy and Chemistry) and Curtin University (MBA), Western Australia.
- I am a Chartered Professional member in good standing of the Australasian Institute of Mining & Metallurgy (MAusIMM (CP)). My relevant experience is over 25 years in mineral processing, process engineering and extractive metallurgy.
- 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.
- I am responsible for the preparation of sections 13, 17, of the Technical Report and parts of section 1 21, 25, and 26 that are pertinent to Metallurgy and Processing.
- I am independent of the issuer as defined in section 1.5 of the Instrument.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
- 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, Western Australia, on 31 October 2019,

"Original Signed"

Simon Walsh

B Sc (Extr. Met.& Chem), MBA, MAusIMM (CP), GAICD
Principal Metallurgist
Simulus Pty Ltd