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VALE INCO LIMITED

EXTERNAL AUDIT OF MINERAL RESERVES

VOLUME 2, SECTION 4

PT INCO OPERATIONS, SOROWAKO PROJECT AREA

Submitted to:
Vale Inco Limited
2060 Flavelle Blvd, Sheridan Park
Mississauga, Ontario
Canada L5K 1Z9

REPORT

Project Number: 10-1117-0032 Phase 4000

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Executive Summary

Golder Associates Pty Ltd (Golder) representatives Mr Iain Cooper, Principal Mining Engineer and Mr Ian Lipton, Principal Geologist, from Golder visited the site from July 5th to 10th, 2010 to carry out an independent audit of the mineral resources and mineral reserves estimated by Vale for the PT Inco (PTI) Sorowako Project Area (SPA) in Indonesia.

During the site visit they inspected mining operations, interviewed personnel and gathered information required to evaluate the appropriateness of the data and methodology used to estimate the resources and reserves. They were accompanied by Mr Tim Lloyd, Principal MRMR Resource Geologist of Vale-Inco. A list of people contacted for this study includes:

- Robbie Rafianto: Manager MRI Planning and Resource Assessment
- Sorimuda Pulungan; Manager Mine Engineering
- Sopi Hakim
- Prakasa Ardiyanto (Anto),
- Sudarmin
- Didi Wahyudi, Superintendent Long Term Planning
- Heru Hariyadi, Superintendent Geotechnical and Hydrology
- Gunawan Marbun, Mining Engineer, Long Term Planning
- Deni

Independent verification of the electronic data was carried out by Mr Richard Gaze, Principal Geostatistician and Gustavo Pilger, Senior Geostatistician of Golder.

This study includes a review of technical reports, memoranda and supporting technical information obtained from PTI. Reports on previous internal and external technical reviews and audits were also made available to Golder (e.g., an independent audit by AMEC carried out in 2007).

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

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

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

Estimated Mineral Reserve for the Sorowako Project Area as of June 30, 2010

Area	Category	Mt DKP	%Ni	%Fe	$\%SiO_2$	%MgO
	Proven (2009 MRMR)	52.9	1.88	23.72	32.42	12.17
	Probable (2009 MRMR)	10.2	1.85	21.31	33.31	13.03
Sorowako West Block	Proven+Probable (2009 MRMR)	63.1	1.88	23.33	32.57	12.31
	Proven+Probable (mined July09 - June10)	*9.2*	*1.91*	*22.42*	*33.46*	*12.54*
	Proven+Probable (remaining June 2010)	**53.9**	**1.87**	**23.49**	**32.42**	**12.27**
	Proven (2009 MRMR)	24.0	1.75	22.46	28.22	13.96
	Probable (2009 MRMR)	5.3	1.66	21.10	30.76	13.85
Sorowako East Block	Proven+Probable (2009 MRMR)	29.3	1.73	22.22	28.68	13.94
	Proven+Probable (mined July09 - June10)	*2.8*	*1.66*	*20.61*	*34.50*	*14.85*
	Proven+Probable (remaining June 2010)	**26.5**	**1.74**	**22.39**	**28.05**	**13.84**
	Proven (2009 MRMR)	11.2	1.83	16.98	34.16	20.48
	Probable (2009 MRMR)	36.7	1.65	14.70	33.90	20.32
Petea	Proven+Probable (2009 MRMR)	47.8	1.70	15.23	33.96	20.36
	Proven+Probable (mined July09 - June10)	*9.5*	*1.67*	*15.17*	*34.50*	*21.15*
	Proven+Probable (remaining June 2010)	**38.3**	**1.70**	**15.25**	**33.83**	**20.16**
Limonite	**Proven+Probable (remaining June 2010)**	**0.4**	**1.38**	**40.80**	**10.39**	**1.68**
TOTAL	**Proven+Probable (remaining June 2010)**	**119.0**	**1.79**	**20.64**	**31.83**	**15.13**

Note: Depleted reserves based on MRMR 2009 - Actual Production from July09 to June 10

Significant Opinions

- ***Golder is satisfied that PTI has met all legal obligations and accordingly considers there is no impediment to the declaration of a mineral reserve. However, given the complex conditions of the CoW, the recent changes to the Mining Law there is some risk to PTI's security of tenure and ability to operate the SPA effectively. PTI is managing this risk by on-going discussions with relevant government agencies.***

- ***Based on the analysis for the QAQC data from SPA, sampling preparation and assaying at SPA are of industry standard suitable for use in mineral reserve estimation and has acceptable errors of precision and no significant bias can be observed. Considerable improvements in cross sample contamination have been made since the 2008 Audit (AMEC, 2009).***
- ***The general approach for estimation in saprolite of using accumulations is supported and correctly accounts for the support effect of the size fraction grades and their corresponding dry weights.***
- ***The overall procedure of applying the economic, geographical, operational and environmental constraints to the mineral resources before they can be considered for the mineral reserves is supported.***
- ***The mineral reserve modifying factors have been developed over a number of years and are reasonable.***
- ***The mining method has been developed and improved over the life of the mine. Selective mining, closure and rehabilitation of mined out areas is an integral part of the mining method. The objective of the selective mining is to ensure that the blend parameters are met. The mining is well supervised.***
- ***In both cost and pricing assumptions scenarios used (Vale and three-year moving average), positive project economics support conversion of mineral resources to mineral reserves. Under sensitivity analysis, in all cases tested, the NPV remained positive, suggesting robust project economics.***
- ***The PTI mine life takes into consideration the new mining law and accordingly the current mineral reserve does not report mineralized material beyond 2035.***

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4.0 PT INCO (PTI) SOROWAKO PROJECT AREA, INDONESIA

4.1 Location

The Sorowako Project is located on the island of Sulawesi in Indonesia (Figure 4-1). The Sorowako operation is situated near the village of Sorowako, approximately 50 km from the coastal town of Malili and approximately 600 km from the city of Makassar in South Sulawesi. Access from Makassar to Sorowako is through a paved road. Sorowako has an asphalt airstrip and is accessible by small fixed-wing aircraft.

Figure 4-1: Sulawesi, Indonesia

The area has a typical wet and humid tropical climate. The temperature ranges from 19 - 30°C. Annual precipitation is about 3,000 mm. Mining and processing operate throughout the year. Mining operations may cease temporarily for short periods during extreme downpours.

The Sorowako Project Area (Figure 4-2) straddles the eastern part of Lake Matano. The rolling hills of Sorowako area (south of Lake Matano) range in elevation from 400 to 800 m. Immediately to the west of the Sorowako area, where ultramafic rocks come in contact with Mesozoic sediments, elevations reach 1,200 m. Numerous basins and sinkhole type features, and plateaus such as plant site area are situated within ultramafic terrain. A flat plain is present near the village of Sorowako. Much of the Sorowako area drains to the north, into Lake Matano, through Tapulemo, Lawewu and Lamoare creeks. The eastern part of Sorowako area drains directly into Lake Mahalona and the Petea River.

Greater relief characterizes the Petea area with elevations ranging from 400 to 1,100m. The Petea area consists of two to three sets W-NW trending terraces that slope to the S-SW. The western part of the area drains directly into the eastern part of Lake Matano through several small creeks, while the eastern part of the Petea area drains into Petea River and Lake Mahalona.

The most prominent topographic feature in the area is the graben-like structure of Lake Matano. The long narrow lake is approximately 600 m deep and represents part of the regional Matano Fault Zone. The Project Area is heavily forested with typical Indonesian rain forest vegetation. The vegetation, particularly the undergrowth, is heavier in the non-ultramafic terrain due to more fertile soil.

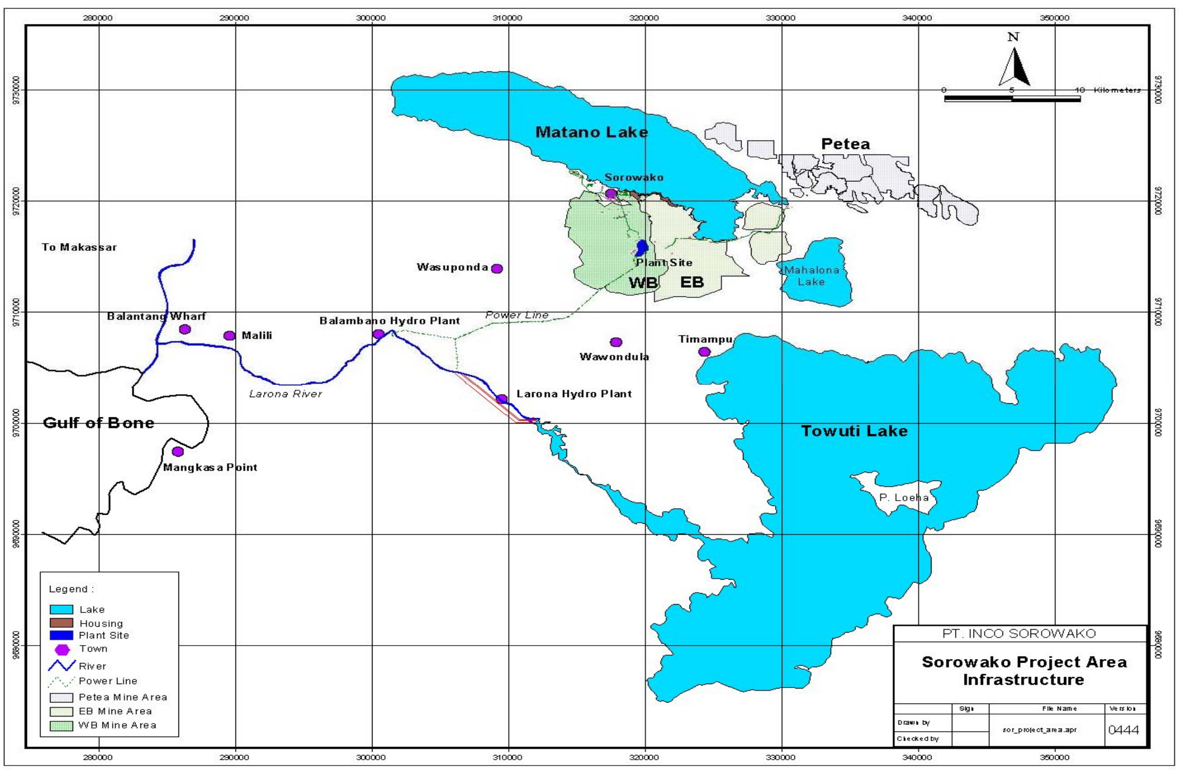

Figure 4-2: Site location of the Sorowako Nickel Project (PTI)

4.2 Ownership

PTI is a publically-traded company in the Indonesian stock market. Vale bought Inco in 2007, gaining its equity interest of 58.73% in PTI, with Sumitomo Metal Mining Co. Ltd (Sumitomo) holding slightly more than 20% and public and other shareholders holding the balance.

4.3 Land Tenure and Mining Rights

To be able to declare a mineral resource and mineral reserve, PTI must hold title to appropriate leases enabling them to continue to operate the mine.

PTI's Contract of Work (CoW) area in Sulawesi consists of 218,529 hectares divided into fourteen separate blocks (Table 4-1). Thirteen blocks are retained for laterite potential while one block (Latao area) is held for chromite potential.

Table 4-1: PTI's Concession Areas in Sulawesi (Kroll *et al*, 2009b)

Province	Concession Block	Hectare
Central Sulawesi	Kolondale	4,512.35
(Total : 36,635.36 Ha or 16.8%)	Bahodopi	32,123.01
South Sulawesi	Sorowako-Towuti	108,377.25
(Total : 118,387.45 Ha or 54.2%)	Matano	6,176.48
	Bulubalang	2,249.33
	Lingke	1,584.39
Southeast Sulawesi	Latao	3,148.11
(Total : 63,506.18 Ha or 29.1%)	Matarape	1,679.87
	Lasolo	4,086.87
	Totobulu	13,817.05
	Pomalaa	20,286.19
	Paopao	6,785.75
	Suasua	10,372.68
	Malapulu (Kabaena Island)	3,329.66
	Total	218,528.99

The Contract of Work areas fall into three administrative provinces: Central Sulawesi (16.8%), South Sulawesi (54.2%), and Southeast Sulawesi (29.1%). The Sorowako Project Area is part of the Sorowako-Towuti concession block in South Sulawesi. A concession map for the Contract of Work is shown in Figure 4-3.

The initial area for the Sorowako Project consisted of 10,010 hectares in East and West Blocks. Exploration and mine development has extended beyond this area to cover a large area of Petea to the northeast.

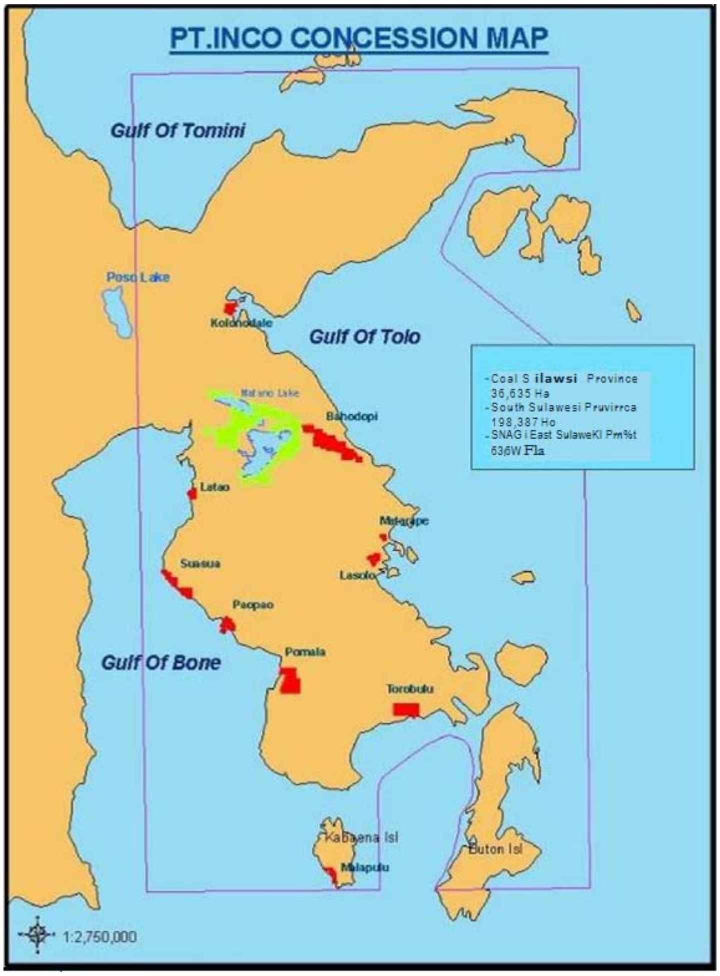

Figure 4-3: Concession Map for Contact of Work (PTI)

The stated information in the MRMR 2009 report by PTI is:

"The original concession of 6.6 million hectares covering the Eastern and South-Eastern arms of Sulawesi was granted on July 27, 1968. The current concession area has been reduced to 3.3% of the original concession through nine area relinquishments, the last of which was in 1979. Commercial production commenced on April 1, 1978.

Under the original Contract of Work or concessionary agreement between the Government of the Republic of Indonesia and PT Inco entered into in 1968 a 30-year production period was granted until March 31, 2008. The Contract of Work was modified and extended by the Agreement on Modification and Extension entered into in January 1996 (the "Contract of Work"). The Contract of Work is valid until December 28, 2025.

PT Inco, as the sole contractor of the Government in the areas covered by the Contract of Work, has been granted exclusive rights in specified areas on the Island of Sulawesi to explore, develop, mine, process, store, transport and sell all nickel and nickel-containing minerals in any form and all minerals found in association with nickel in the Contract of Work areas. The Contract of Work also grants PT Inco all necessary licenses and permits to conduct its operations, including certain expansions of its operations, as provided for in the Contract of Work. However, PT Inco has been required to secure permits and licenses in certain circumstances from various government ministries and various levels of government. Additionally, the

Government may disapprove plans for construction, operations or expansions based on certain limited grounds regulated in the Contract of Work.

Indonesian Law No 41 of 1999 restricts open pit mining and certain other activities within areas designated as "protected forest", without incorporating transitional provisions with respect to pre-existing mining contracts and licences, such at PT Inco's Contract of Work. A portion of the area PT Inco is authorized to mine under its Contract of Work is considered to be protected forest.

Under the Contract of Work, PT Inco agreed to construct production plants at Pomalaa in South East Sulawesi and Bahodopi in Central Sulawesi, subject to economic and technical feasibility. The Contract of Work indicated that the first plant could be in operation by about 2005 and the second by about 2010, but did not specify which plant was to be constructed first.

PT Inco discontinued its ore supply to PT Aneka Tambang Tbk (PT Antam) for its FeNi III plant in Pomalaa as of July 18, 2008 following expiration of the Cooperative Resources Agreement (the "CRA"). With the discontinuance of ore supply to PT Antam and based on the letter received from the Minister of Energy and Mineral Resources, PT Inco was required to deliver a report by April 2009 evaluating the economic and technical feasibility of the construction of a high pressure acid leach processing facility at Pomalaa to produce nickel hydroxide, an intermediate nickel product, with an annual production capacity of approximately 30,000 tonnes. The report concluded that, under the current assumptions, the construction of such a processing plant is not technically and economically feasible at this time. PT Inco requested an additional 2 year period to carry out a further study. On June 23, 2009, the Government responded requiring PT Inco to return the further study at the latest by the end of December, 2009.

In addition, since the processing plant at Pomalaa is not currently technically and economically feasible, a refinery facility at Bahodopi with an annual production capacity of approximately 30,000 tonnes to process nickel hydroxide from Pomalaa is not currently feasible either (unless there is an alternative source for its feed).

It should be noted that the development of any project is subject to agreement with relevant levels of the Government on assurances and incentives that support the significant capital investment involved and satisfy return of investment considerations, including certainty with respect to the term of the Contract of Work.

The Indonesian Parliament passed a new Mining Law superseding the 1967 Mining Law on December 16, 2008. The new Mining Law was promulgated and became effective on January 12, 2009. It sets out a broad regulatory structure and provides that many important details will be clarified in later implementing regulations that should be issued within one year of its effective date (that is, at the latest by January 12, 2010). Draft regulations have been prepared by DEMR. The impact of the new regulations on PT Inco's existing Contract of Work will remain unclear until the implementing regulations are passed."

Golder is satisfied that PTI has met all legal obligations and accordingly considers there is no impediment to the declaration of mineral reserve. However, given the complex conditions of the CoW, the recent changes to the Mining Law there is some risk to PTI's security of tenure and ability to operate the SPA effectively. PTI is managing this risk by on-going discussions with relevant government agencies.

4.4 Infrastructure

Kroll *et al* (2009a) summarises the plant infrastructure in the SPA, which consists of:

- Two hydro projects generating a total of 323 MW of power.
- Diesel and steam generators with a total capacity of 46 MW.
- A landing strip at the village of Sorowako.
- Hospital, town sites, roads and school.

- Oil storage and wharf facilities at the port of Malili about 70 km from Sorowako.
- Three mining areas designated East Block, West Block, and Petea.
- An integrated mineralised material drying, reduction smelting and converting facility.
- Offices and service facilities.
- Potable and process water are obtained from Lake Matano.
- Appropriate areas designated for tailings disposal, mine waste disposal, wet ore stockpiles, and settling ponds.

The mining operation has been running successfully since 1978. The infrastructure is appropriate for its needs.

4.5 Production Process and Products

The following overview is reproduced from Kroll *et al* (2009a, 2009b).

The mineral reserves for the Soroako Project Area (SPA) consist of deposits in three areas; West Block, East Block and Petea. The process plant is located at Harapan, between the West and East Blocks and was fed exclusively with mineralised material from these areas until 2005. Mining began at Petea in 2005, replacing the mineralised material feed from East Block.

The deposits are typically delineated in separate hills and the mineral resources and mineral reserves in each hill may be split into several models. For mine planning the hills are divided into compartments and each compartment has its own mine plan and schedule. The compartments system is used primarily to manage the need to backfill waste into mined out areas and minimise waste haulage distances. There are no external waste dumps.

PTI in SPA currently is operating in two mining areas based on mineralised material type: West Block mine for West type mineralised material and Petea mine for East type mineralised material. Petea, a new mining area began production in 2005 and is expected to continue to be mined throughout the life of the Sorowako Project to replace the conventional East type material in the East Block. Although Petea is a new location for mining, the mineralisation type is similar to the highly serpentinized variety of saprolite originally found in the East Block. Remaining mineralisation in the East block, on the other hand has been re-evaluated as –1” high olivine material; a west type material.

The surface mining is not traditional pit mining but more an “open cast” system by truck and shovel where contour side cuts are made into a hill. Anywhere from eight to ten hills are actively mined to provide run-of-mine (ROM) that meets the quality specifications (ore chemistry) of feed for the nickel processing plant.

ROM is delivered to one of five Screening Stations. The screen size varies according to the type of mineralisation being delivered; a size of -2” product for West Ore and -6” product for East Type mineralisation. Some mineralisation is also crushed at the screen stations. Screen Station Product (SSP) is trucked to the Wet Ore Stockpile (WOS). The WOS consists of 29 separate compartments for mineralisation of different chemical composition. The compartments are built on a sloping base in which there are drainage channels filled with permeable gravel and covered with geotextile and coarse rock. The drainage channels are designed to promote dewatering off the stockpile but in practice they become blocked after some time and less effective. Plant feed material resides in the WOS for at least six weeks in order for some drying to occur before a blended feed is then supplied to the dryer kiln for complete drying. WOS compartments are operated on an open-fill- close, empty cycle, so the grade of the compartment is well-known and the blend to the dryer kiln can be managed.

The product from the rotary kiln is termed the Dryer Kiln Product (DKP). The DKP is placed in Dry Ore Stockpiles (DOS) prior to reclaiming and feeding to the Reduction Kilns and Furnaces. Six stockpiles are currently maintained in two large sheds. Ore from the DOS is then blended and fed to the kiln at the front end of the matte plant.

It is extremely important to control the quality of mineralisation fed to the matte plant within narrow chemical limits. This control is achieved by extensive sampling at the mine face, the screen stations and the dryer kiln product, and by careful blending of the mineralisation through the screen stations, WOS and DOS. .The target chemistry for the LOM plan for SSP is 20% Fe and a silica/magnesia ratio of 2.1, which PTI anticipates will be applicable due to improvements from changing of West screening stations from 4” to 2” recovery.

Nickel Processing

Golder reviewed the processing as described in Kroll *et al* (2009a, 2009b) and from discussions while on site. A summary process flowsheet is provided in Figure 4-4, which indicates that the process plant is a complex operation that involves numerous phases of processing.

The pyrometallurgy nickel processing used at PT Inco involves in initial mineralisation drying process, followed by reduction, smelting and converting to produce the final dried product.

Drying - Wet mineralisation from the WOS is fed to three rotary kiln dryers. This process partially dries the feed mineralisation from a moisture content of 30 to 40% to produce DKP, with a reduced moisture content of 19 to 21%. East and West type mineralisation are separately screened to produce -1” DKP, with the product of each type mineralisation placed in separate piles in Dried Ore Storage buildings.

Reduction - From the Dried Ore Storage, mineralisation is fed to the counter-current fired reduction kiln. A blend of East and West type mineralisation is used. The reduction kiln comprises three zones with increasing temperatures and reduction potentials. The first zone pre-heats the DKP feed with 19 to 21% moisture, completely drying it. The second zone, referred to as the calcination zone, removes the water of crystallisation. The third zone, referred to as the reduction zone, contributes to most of the partial reduction, which is achieved by the injection of High Sulphur Fuel Oil. Sulphur is introduced and combines with the iron and nickel to form iron nickel sulphides. The product from the reduction kiln is called calcine. Calcine mineralisation overflows the discharge end of the reduction kiln into a refractory-lined surge bin and transported to the furnace feed bins.

Smelting – The calcine is smelted in the electric furnaces operating in an open arc mode. Here, the oxide (slag) phase separates from the sulphide (matte) phase. The slag is skimmed from the furnace almost continuously and is disposed. The matte is tapped periodically as required by the converters. The assay of the furnace matte is in the range of 25-28% Ni.

Molten furnace matte is transferred to the converters through ladles. Air/oxygen is blown in to oxidise the remaining iron. Silica flux is added to flux the oxidised iron that is then slagged away. During converting, the lower grade electric furnace matte is converted to Bessemer matte with a nominal analysis of 78% Ni, 20% S and 2% Co.

Finally, converter product is granulated, dried, screened and packed in bags for shipment. PTI is paid 78% of the prevailing London Nickel Exchange (LME) nickel price. Current annual production is about 170 million lbs of nickel in matte. A decision has been made to construct a third dam on the Larona River as part of a capital expenditure program intended to increase production to about 200 million lbs of nickel in matte per year.

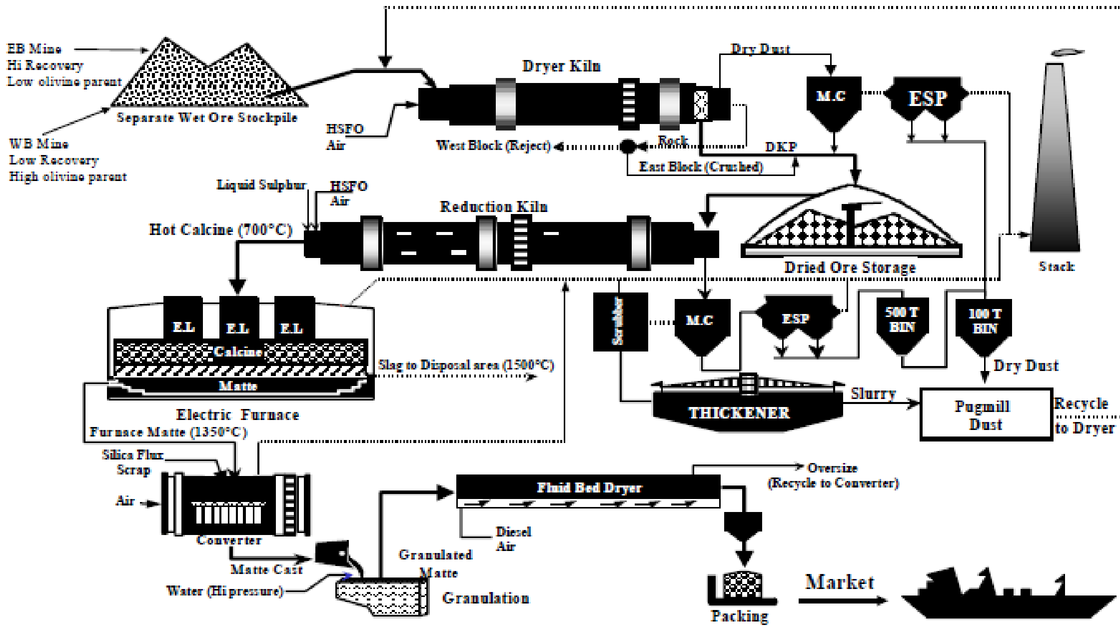

Figure 4-4: Process Flow Sheet for the PT Inco Plant (from Kroll et al, 2009b)

Operational Considerations

Golder notes from Kroll *et al* (2009b) the following aspects from nickel processing:

Initial smelting of West Block mineralisation yielded a highly corrosive slag, causing erosion of the refractory lining at full power. The silica/magnesia ratio of 2.2 to 2.4 of the slag was too acidic for the basic refractory furnace, combined with a liquidus temperature which was too low.

PTI mitigated this by using:

- a higher superheat in the feed to prevent slag deposition on the refractory walls
- introducing more basic East Block type mineralisation feed and blending with West Block type mineralisation to achieve an acceptable liquidus temperature.
- Creating a protective layer of slag against the furnace refractory by replacing the cooling of the furnace shell by falling water to the use of water cooled copper fingers surrounding the slag zone.

These changes successfully stabilised the furnace operation, although at the expense of higher power consumption and lower feed grades. Some improvements to control of feed chemistry and tolerance of mineralisation acidity has reduced the proportion of East Block type mineralisation in the feed over the last few years. This has yielded a higher feed grade and increase in the production of nickel matte.

4.6 Metal Recoveries

The process plant nickel recovery used to estimate the mineral reserves is 86.3% based on the average process plant results for the period 2004-2009. An increase in the process plant recovery, from 85% in 2008 to 88.9% in 2009, was achieved due to improvements in the unaccounted nickel loss calculations. For the purposes of the LOM financial analysis, the five year average recovery of approximately 86% was used.

4.7 Market

Golder has reviewed the following information from Kroll *et al* (2009a) and considers it to be reliable and factual.

All of PTI's production is sold under long-term contracts to Inco and Sumitomo, which will continue until the expiration of the Contract of Work. These contracts provide that if the Contract of Work is extended or renewed, the contracts will be extended for the period of the extension or renewal.

Currently PTI produces a nickel in matte. The nickel content and impurities may vary, over time, depending on mineral distribution and association within the deposit, processing techniques and other factors. The end product will be in a form that is expected to be used directly for the production of stainless steel.

Primary nickel demand for use in stainless steel production represented about 60% of world nickel demand in 2007 or approximately 815,000 tonnes of nickel.

4.8 Historic Production

Table 4-2 lists mine production, based upon PTI's Dry Kiln Product (DKP), and the average grade for nickel at PTI for the three main mining blocks from 1976 to 2009. The DKP is subject to royalty payments.

The mine production strategy of PTI is to provide a feed blend that optimizes process plant operation and throughput. Current chemistry targets are to supply SSP blend @ 20.0% Fe, and a Si/Mg ratio of 2.10.

Test production in 1976 encountered problems with high acid West Block mineralisation, addressed with blending with less siliceous East Block mineralisation. Full production commenced April 1978 and produced nickel matte averaging 78% Ni, 2% Co and 20% S.

Table 4-2: Historic Mine Production in DKP (in millions of dry tonnes)

West Block DKP (1976 – 2009)		East Block DKP (1979 – 2005)		Petea DKP (2006 – 2009)		Total DKP (1976 – 2009)	
Mt (DKP)	% Ni	Mt (DKP)	% Ni	Mt (DKP)	% Ni	Mt (DKP)	% Ni
40.8	2.04	32.1	1.76	5.5	1.84	78.4	1.91

4.9 Geology and Mineral Deposits

Regional Geological Setting

Regional, local and mine geology is well understood by Vale personnel and this understanding is of a sufficient level for the reporting of mineral reserves. The following description is taken from PTI reports:

"The region has been subdivided into four lithotectonic units bounded by largescale tectonic dislocations and thrust faults. These are from west to east: (i) the west Sulawesi volcano-plutonic Arc, (ii) the central Sulawesi metamorphic belt, (iii) the east Sulawesi ophiolite belt, and (iv) the continental fragments of Banggai-Sula, Tukang Besi and Buton.

The East Sulawesi Ophiolite (ESO) is a dismembered ophiolite that is tectonically intercalated with Mesozoic deep-sea sediments, and probably includes Indian Ocean MORB, marginal basin crust, and parts of the Sundaland fore-arc or oceanic plateau of Pacific plate. The ESO is one of the three largest ophiolites in the world. The total length of the ESO is some 700 km from Gorontalo Bay, through the East Arm and central Sulawesi toward the Southeast Arm and the islands of Buton and Kabaena; it also extends to the Lamasi complex of the South Arm passing through the Gulf of Bone. The ophiolites are intercalated and complexly juxtaposed with Mesozoic and Tertiary sedimentary rocks, as a result of late Oligocene/ early Miocene collision, subsequent contraction, and later strike-slip faulting.

The Sorowako ultramafic complex is part of the large Lakes Area ultramafic massif that extends from the eastern coast of Sulawesi to the western end of Matano Lake. Locally, the complex is bounded in the west by a west-dipping thrust fault that separates it from Mesozoic sediments and by Tertiary sediments in the southeast. The complex is subdivided into four different ultramafic domains, i.e., West block, East Block, Petea and ultramafic conglomerate.

West Block (Unserpentinized Hazburgite) are typically harzburgites as they contain olivine (average 80-90%) and orthopyroxene (average 10-20%). Minor amounts of subhedral chromite, generally 1% or less, are observed. Traces of clinopyroxene are locally encountered. A few occurrences of dunites with less than 90% olivine and minor amounts of subhedral chromite are also observed. The thin serpentine fracture fillings seen in thin section correspond to the film-like cover of soft serpentine on broken surfaces of megascopic samples.

East Block (Serpentinized Lherzolite) are typically Iherzolites that contain olivine (average 60-65%), orthopyroxene (average 25-30%) and clinopyroxene (average 10%). Minor amounts of anhedral chromite, generally 1% or less, are observed. A few occurrences of harzburgites and dunites are also present. Two of the three dunites are highly serpentinized and the original mineralogy is interpreted to be 99% olivine. The rocks of the East Block are variably serpentinized. Samples from the eastern part of East Block are weakly to moderately serpentinized while samples from the western part of East Block to the West Block are strongly serpentinized and are referred to as the traditional East.

Ultramafic Conglomerate (Ultramafic Sediment) Conglomerate and greywackes are found over considerable areas and have been deposited unconformably above ultramafic bedrock. The coarse fraction consists totally of ultramafics. The conglomerate tends to have a porous sand matrix that often is filled by supergene silicate precipitates (including garnierite). The boulders and cobbles tend to be well rounded, imbricated ellipsoid commonly serpentinized and almost invariably serpentinized along their margins. The sand fractions in greywacke are mostly subrounded serpentine particles or partly serpentinized pyroxene and amphibole. The pore space is filled with isotropic serpentine. The contact between peridotite and conglomerate are irregular and sharp and many authors interpret the conglomerate as deposited after emplacement of the ultramafic complex.

Petea (Serpentinized Hazburgite and Lherzolite) is a series of potentially favourable laterite landforms extend for a distance of 30 km from the eastern shore of Lake Matano to Lampesue area in the east. The Tanamera and Petea area, located in the west, comprise approximately half of this large prospective area. In the Petea area exploration of the ultramafic rocks to date indicate the lithologies consist essentially of harzburgite and lherzolite that have been moderately to highly serpentinized. Unsepentinized peridotites

(West Block type) are not present in the area. Almost 95% of all systematic rock samples analysed from core holes and test pits indicated olivine contents of less than 20%. The Petea area has undergone extensive secondary tectonism due to the Matano Fault. This secondary faulting is responsible for localising the drainage in the area and has hastened the development of lateritic soils by providing adequate channels for ground water movement. The laterite profile developed at Petea is very similar to profiles developed in the Sorowako area. On average, the overburden at Petea is thinner and grades are comparable to true East type material previously mined at Sorowako."

Mine Geology

Nickel laterite deposits in the Sorowako and Petea area are generally classified into two main types, West Block and East Block types, depending upon the nature of bedrock, the level of serpentinisation, ease of mining, size of boulders, degree of fracturing of the bedrock, optimum screen fraction that carries nickel grades, and the silica-to-magnesia ratios in the mineralisation. Figure 4-5 shows the generalised laterite profile.

The following description is taken from PTI reports:

"West Block type deposits

These deposits are developed over a relatively unserpentinised peridotite bedrock; commercially mineable grades are confined to the -1" fines fraction; and silica to magnesia ratios are high (2.2 to 2.6 range).

West Block type deposits are categorized into Types 1, 2 and 3, depending on the degree of fracturing in the bedrock and the size of the resulting boulders. Type 1 deposits are the most difficult to mine due to weak fracturing of the bedrock and the resulting large size of the boulders. Type 2 and Type 3 deposits are successively easier to mine and are formed over more fractured bedrock resulting in boulders within the mineralisation profile that mining equipment can handle.

Optimum nickel grades in all West Block type deposits are confined to -1" screen fraction. Oversize is generally barren or very low grade. Exceptions exist in breccia mineralisation where siliceous garnierite carries respectable grades in the +1" fraction.

East Block type and Petea deposits

These deposits are developed over bedrock that is much more serpentinised and characterized by generally low silica to magnesia ratios. They are further categorized into three types based on upgrading characteristics of the mineralisation as follows:

18" mineralisation type in which all fractions up to 18" size are mineralisation grading. This is also referred to as "Pure East Type mineralisation".

6" mineralisation type in which only material less than 6" in size is mineralisation grading; oversize rock is low grade and needs to be rejected.

1" mineralisation type in which only -1" material is mineralisation grading and all oversized material is waste. This mineralisation type is further divided into two types depending on the olivine content of the +1-6" size fraction:

- *Low-olivine -1" mineralisation type in which the olivine content of the +1-6" rock is generally less than 22%. This mineralisation type can be screened at 1" size for maximum nickel upgrading, or screened at 6" size for maximum mineralisation recovery. In either case olivine content is acceptable for the electric furnaces.*
- *High-olivine -1" mineralisation type in which the olivine content of the +1-6" fraction is generally more than 22%. This mineralisation type must be screened at 1" size to ensure that excessive olivine does not enter the electric furnaces.*

East Block 6"/18" mineralisation type form an arcuate zone extending from Sumasang in the north to Lamangka South and Fiona-Farah in the south-east. East of this zone, and up to the shore of Lake Matano, the area is underlain largely by -1" high-olivine mineralisation type. East Block -1" low-olivine mineralisation type only forms irregular patches within the 6"/18" mineralisation type and the -1" high-olivine mineralisation type.

In the Mahalona North area, the 6"/18" mineralisation type has numerous small raisin-like zones of the other two East Block mineralisation types.

Petea area was originally classified as -18" mineralisation type but was not supported after reviewing borehole and test pit data. Recent results from mining (screening results) indicate that Petea is generally a -6" mineralisation type. Petea will continue to be treated as -6" mineralisation. The -1" mineralisation type identified in Petea is treated at -6"."

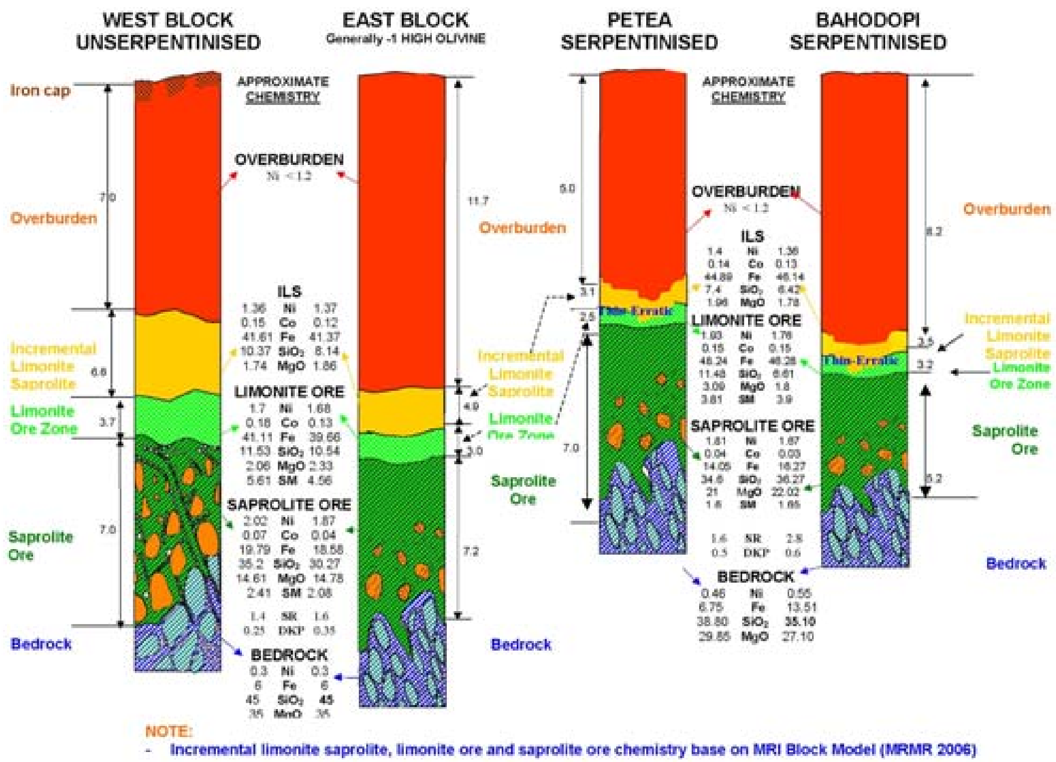

Figure 4-5: Sorowako deposits generalised laterite profile

4.10 Exploration and Development Drilling

Various historical drilling has been conducted over the SPA, as summarised in Table 4-3. Since 2001 core drilling has been used at Saraowaho. Initial sampling in the area used based on a 400 m grid, progressively infilled at 200 m spacing. Targeted mineralised areas with better economic potential are drilled at 100 m spacing.

In the SPA, all early exploration sample sites were surveyed by theodolite. Collar elevations and coordinates for the recently introduced core drilling are picked up by electronic theodolite. PTI's current practice is to carry out DGPS surveying immediately after tree cutting. Revised survey data in case necessary is then used to re-run the block model for final and detailed mine planning and actual grade control. Specific down hole surveying is not undertaken, with all holes are assumed to be vertical. This is a reasonable assumption for this type of deposit.

Table 4-3: Historical Drilling of Sorowako Project Area

Period	East Block Number	Percent	West Block Number	Percent	Petea Number	Percent	Total Number	Percent
Auger Drilling								
2001-2004	17,164	100%	25,185	100%	0	0%	42,349	100%
Core Drilling								
2001-2002	3,103	35.15%	1,342	12.70%	665	19.95%	5,110	22.48%
2003	1,630	18.47%	2,674	25.31%	1,451	43.52%	5,755	25.32%
2004	725	8.21%	2,601	24.62%	716	21.48%	4,042	17.79%
2005	2,248	25.47%	2,293	21.70%	0	0.00%	4,541	19.98%
2006	476	5.39%	1,253	11.86%	6	0.18%	1,735	7.63%
2007	645	7.31%	131	1.24%	221	6.63%	997	4.39%
2008	0	0.00%	234	2.21%	201	6.03%	422	1.91%
2009	0	0.00%	38	0.36%	74	2.22%	129	0.49%
Total Core drilling	**8,827**	**100%**	**10,566**	**100%**	**3,334**	**100%**	**22,727**	**100%**

During the period January to mid-November 2008, exploration and mine development drilling was carried out in the Petea and West Block. A total of 159 holes for 2,970 m was drilled for exploration in Petea Blocks E and F. 42 holes for 948 m was drilled for mine development in Petea Block B North. In the West Block, 132 holes for 277 m was drilled for confirmation purposes in Anoa Valley and Pakalangkai. Mine development drilling was carried out in Songko (27 holes for 855m in) and Nickel Hill (175 holes for 4,934m). Up to end of 2008, the exploration program was re-drill for confirmation purposes in Petea Block B.

Drilling procedures are industry standard and Golder considers them as appropriate for Nickel laterite deposits.

4.11 Deposit Sampling Methods and Data Management

Data Used for Resource Modelling

Early in the history of the SPA, test pitting and auger drilling were commonly used for exploration and definition of mineral resources. It was later recognised that the quality and penetration of these sampling methods was not adequate for the estimation of mineral resources. Since 2007 all mineral resources have been explored for and estimated with core drilling data alone.

Drill Sampling

All of data used in the current resource and reserve models is from core drilling. A diamond drill rig was active at the time of the site visit and was observed as part of this audit (Figure 4-6).

A geologist or geological technician supervises the drilling and logs the core as it comes out of the hole. Triple tube HQ core is drilled using as little additional water as possible, so as to minimise core loss. The core inner tube is pushed out of the core barrel with water pressure and the core is fed from the split inner tube into plastic core trays and the core recovery measured and recorded on a paper log.

As each core tray is filled, the geological technician measures and logs the core recovery, and a simple rock type code (L=limonite, S=saprolite, T=transition and B=bedrock). The driller signs the log sheet. A more detailed log is completed by the geologist using a Trimble PDA. This log includes more detailed description of the rock type, major and minor minerals, grain size, weathering, etc. The degree of serpentinisation is estimated based on a combination of visual examination, magnetic susceptibility (serpentinised material is more magnetic), and LOI data from the laboratory. This information is used to map high-Olivine (HO) material, so that it can be controlled in the blend during mining.

Each drill run is 1m, but a 1.5 m core barrel is used because core recovery commonly exceeds 100% particularly in the limonite zone. This phenomenon is known as 'extrusion' and it was observed during the site visit in hole C19-0-914. A continuous core of limonite of 1.46 m length was recovered from the interval 3-4 m. The core appeared undisturbed. Since there is no evidence of swelling or heave of limonite exposed in the open pits the most likely explanation for the additional core recovery is that very soft material such as limonite, with a consistency of paste, is partly cut by the core bit and partly flows inside the core tube. This occurs when drilling advances faster than the bit cutting action in soft formations. Therefore, in a hole of HQ size, where the outside hole diameter is 96 mm and the core diameter is 63.5 mm, as much as 150% core recovery can be expected if none of the material is cut by the bit face but rather is pushed into the core barrel during drilling.

PTI presented summaries of core recovery data in the 2009 MRMR report. These summaries indicate that over half the limonite samples have core recoveries greater than 100%. In the saprolite, extrusion is less common but still affects about 10-20% of samples. PTI assumes that measured core recovery greater than 100% is indicative of core recovery of 100% and "corrects" the data by reducing core recoveries greater than 100% to 100%. Drilling procedures now include the aim of drilling a minimum of 5 m into the bedrock, however this has commonly not been achieved because true unmineralised bedrock is commonly difficult to recognise.

Figure 4-6: Diamond drilling observed during the Golder site visit

Grade Control

There is no short term or grade control modelling. Consequently, all mineralisation mining is closely supervised by a geologist or experienced technician. As mining approaches the expected top of mineralisation, grade control sampling is carried out on the benches. Samples are collected from polygonal areas around sampling points on an approximate 5 m grid. The sampling nodes are marked with wooden stakes and identified with labels written onto plastic flagging tape. The samples are poured through a 2" screen, mixed on a rubber mat and cone and quartered to collect approximately 2kg of -2" material. At Petea, where there is a significant portion of +2" mineralisation, a +2" and -6" sample is also collected by screening.

The grade controllers also collect a grab sample from approximately every 500 t of material loaded by the shovels. This is taken at random across the top of the shovel bucket.

The samples bagged with a pre-numbered ticket and are analysed at the Mine Rush Assay Laboratory (MRAL). The turn-around time is typically 1 – 2 hours, with the results delivered to the grade controllers by radio or cell-phone. The results of both are used informally by the grade controllers to direct the shovel operators. A simple sample log (date, bench, GPS coordinates, estimated Ni grade and reported Ni grade) is maintained by the grade controllers.

Further sampling of the run of mine mineralisation occurs at the screen stations. A mechanical belt scoop sampler is used (it was not made clear whether this occurs in stop-belt mode) to collect a sample of about 70 kg for every 500t of Screening Station Product (SSP). This material is mixed on a rubber mat, coned and quartered to produce a 2kg sample for the MRAL and the remainder is put in steel buckets and sent to the trommel testing station at Harapan. The trommel station is used to carry out a better controlled screening, drying and sampling process which provides better estimates of the quality of material being delivered from the screen stations to the Wet Ore Stockpiles (WOS).

Sample Preparation

The sampling and sample preparation methods used on site are fully described in Kroll *et al* (2009b) and were demonstrated to Golder on site.

Core Samples

Core samples are prepared in indoor preparation rooms at Enganno and Petea. Golder inspected the facilities at Enganno, which can process 6000 samples per week (Figure 4-7).

All core boxes are checked against the field logs and photographed on arrival at the preparation area. Cores are sampled over 1 m intervals or less if the rock type changes (e.g., if there is a core stone). The core is sampled by size fraction (-1", -2", -6"). None of the core is retained for audit or reference. Coarse material is passed through a two tier screen (2" and 1" mesh) to produce three size fractions. Each fraction is mixed on a rubber mat and then subdivided to approximately 2kg by cone and quartering. The size fractions are placed in separate sample bags. Limonite and other obviously fine grained material is not screened but goes straight to cone and quartering. All samples are labelled and tracked through the preparation process with barcodes.

Figure 4-7: Core sample preparation

The 2 kg samples are then passed through to a second room for drying and comminution. The samples are weighed then dried in a conventional electric oven for 8 hours at 105°C. They are weighed again after drying, to determine moisture content. PTI has carried out tests on the efficiency of drying.

The samples are crushed to 10# (1.6 mm) in a Boyd Crusher. The crushed samples are then mixed by passing them through a riffle splitter three times, then the same riffle is used to reduce the sample mass to around 200- 250 g. The left-hand side splitter tray is always the rejected one.

The sample is then pulverised in a Continuous Ring Mill (CR Mill) for 3-4 minutes to reduce the particle size to 200# (74 microns). Since late 2007 a barren granitoid sample wash has been used in the CR Mill between every sample.

The pulverised samples are stored in paper Kraft bags. A subsample of 20 g is submitted to Proctech for analysis. The subsamples are obtained by rolling the 200g pulp on a fresh paper sheet several times, then taking several scoops in a herringbone pattern with a small spatula.

All steps in the sample preparation are tracked by barcode and the data (e.g. sample numbers, weights, etc) is recorded in the new dataXPLore database. A variety of quality control samples are tested. The QC data is processed weekly. The analysis of QAQC samples is discussed below.

Grade Control Samples

Grade control samples include grab samples from the mine benches and shovel buckets, belt samples from the screen stations, etc. They are collected and processed in a very short time in order to provide information on material movements around the mines and to allow decisions about where to direct material. They are processed at the MRAL. There are MRALs at the mine offices in Harapan and at Petea. Golder inspected the MRAL at Harapan.

The MiniPal results are regularly checked against assays of the same pulps in the ProcTech laboratory. The results of this comparison indicate that the MRAL analyses are adequate for the grade control tasks for which they are being used.

MRAL turnaround is impressive and seems to be a significant factor in the success of the grade control. PTI advised that the main objective of MRAL is the rapidity of sample delivery for mining. PTI's view is that the relative accuracy of sample analysis is acceptable to support daily mine operation and the discrepancy of MRAL and Protech results is controlled regularly.

Assaying

Core Samples

Appropriate documentation is in place to define the protocols to be used for sampling and assaying and these are being routinely applied. External audits are periodically carried out on the sample preparation and assaying.

Data is transferred electronically from the LIMS to the dataXPLore database for review by the Quality Control geologists in the sample preparation area.

Fused bead XRF data is now the primary data used for mineral resource estimation. Samples previously analysed by pressed powder XRF were re-assayed by fused bead XRF between 2006 and 2008.

Quality Control Results (QC)

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

Golder was provided with the QA procedures and the QC data from Sorowako Project Areas (SPA) from July 2009 to June 2010. In the following sections, the QA procedures and the results from the analysis of the QC data will be discussed.

Procedures

Appropriate documentation is in place to define the protocols to be used for sampling and assaying and these are being routinely applied. Independent external audits are routinely carried out on the sample preparation and assaying (e.g. Djafar and Kadarusman, 2009; AMEC, 2009).

- QAQC procedures at SPA include the insertion of a variety of quality control samples. Regular reviews are carried out on the analysis of QAQC samples and the QC data is processed weekly.

Standard Sample

Accuracy of assaying is normally monitored independently of the laboratory checks, using Standards, (also known as Certified Reference Materials or CRMs but at Sorowako the acronym CRM refers to the Continuous Ring Mill). These are usually samples of pulp material of known (usually certified) grade, which are submitted to monitor the accuracy of a laboratory (the ability of the laboratory to get the correct or known result). Making the Standards from drill sampling residues from the SPA produces Standards with the same matrix as the mine samples.

PTI has been submitting five standard samples as listed in Table 4-4 between 2008 and 2010. Golder carried out checks on major elements for samples submitted since the last external audit (AMEC, 2009), i.e. Ni, Fe SiO2 and MgO values on samples submitted between 2009 and 2010.

Table 4-4 details the acceptable mean and standard deviation supplied by PTI.

Table 4-4: Standard Samples submitted and the accepted values

Std Id	No. of Sample	Elements	Ni	Fe	SiO2	MgO
OB	784	Mean (%)	0.619	46.200	6.720	4.800
		St Dev	0.010	0.470	0.060	0.070
MGL	775	Mean (%)	1.305	43.970	10.980	2.510
		St Dev	0.024	0.420	0.200	0.060
SNORE	939	Mean (%)	0.960	13.930	51.870	17.500
		St Dev	0.008	0.060	0.520	0.210
SORE	1133	Mean (%)	2.810	13.880	40.850	23.250
		St Dev	0.030	0.140	0.320	0.220
BRK	624	Mean (%)	0.477	7.640	41.390	41.490
		St Dev	0.013	0.100	0.370	0.340

The result of Golder's analysis of the standard sample data is summarised in Table 4-5 and detailed in the following paragraphs.

Table 4-5: Result of the analysis on the standard samples

	Sample ID	Outliers% (Mean ±3σ)	HRD%	HARD%	No. Samples	Comment
Ni	OB	6.76%	0.12%	1.13%	784	Excellent precision and accuracy
	MGL	1.94%	-0.38%	0.86%	775	Excellent precision and accuracy
	SNORE	26.94%	-0.71%	0.96%	939	Excellent precision and accuracy
	SORE	5.30%	-0.45%	0.71%	1133	Excellent precision and accuracy
	BRK	2.56%	0.55%	1.05%	624	Excellent precision and accuracy
Fe	OB	2.93%	0.29%	0.61%	784	Excellent precision and accuracy
	MGL	4.26%	-0.27%	0.63%	775	Excellent precision and accuracy
	SNORE	35.57%	-0.33%	0.64%	939	Excellent precision and accuracy
	SORE	1.77%	0.001%	0.54%	1133	Excellent precision and accuracy
	BRK	1.60%	-0.07%	0.84%	624	Excellent precision and accuracy
SiO2	OB	29.85%	-0.83%	1.15%	784	Excellent precision and accuracy
	MGL	1.16%	-0.24%	0.86%	775	Excellent precision and accuracy

	Sample ID	Outliers% (Mean ±3σ)	HRD%	HARD%	No. Samples	Comment
	SNORE	2.34%	-0.02%	0.49%	939	Excellent precision and accuracy
	SORE	5.21%	-0.29%	0.48%	1133	Excellent precision and accuracy
	BRK	6.25%	-0.55%	0.62%	624	Excellent precision and accuracy
MgO	OB	2.55%	-0.01%	0.87%	784	Excellent precision and accuracy
	MGL	4.90%	-0.75%	1.63%	775	Excellent precision and accuracy
	SNORE	1.60%	-0.24%	0.47%	939	Excellent precision and accuracy
	SORE	2.47%	-0.39%	0.49%	1133	Excellent precision and accuracy
	BRK	3.53%	-0.73%	0.74%	624	Excellent precision and accuracy

Note: The analysis of the standard samples shows excellent accuracy (-0.83 to 0.55% HRD) and precision (0.47 to 1.63% HARD).

Blanks

Blank or barren samples are materials with very low expected Ni grade. These are submitted to ensure that there is no contamination between samples during the sample preparation or assaying. If the blank samples following high-grade samples have elevated grades then there is an indication of problems. The Limit of Detection (LOD) is defined as the lower limit of assaying where the precision approaches ±100%.

In the previous external audit (AMEC, 2009) concern was raised on cross sample contamination. A comparison of the 2008 and 2010 data shows significant improvement. While there are still 2.46% of the samples have a Ni values above the upper limit of 0.047%Ni (Figure 4-8), the concentrations of Ni in these blank samples are lower compared to 2008 (max. 2008 Ni = 0.39%, max. 2010 Ni = 0.11%). All samples show satisfactory results for Fe, SiO_2 and MgO (Figure 4-9).

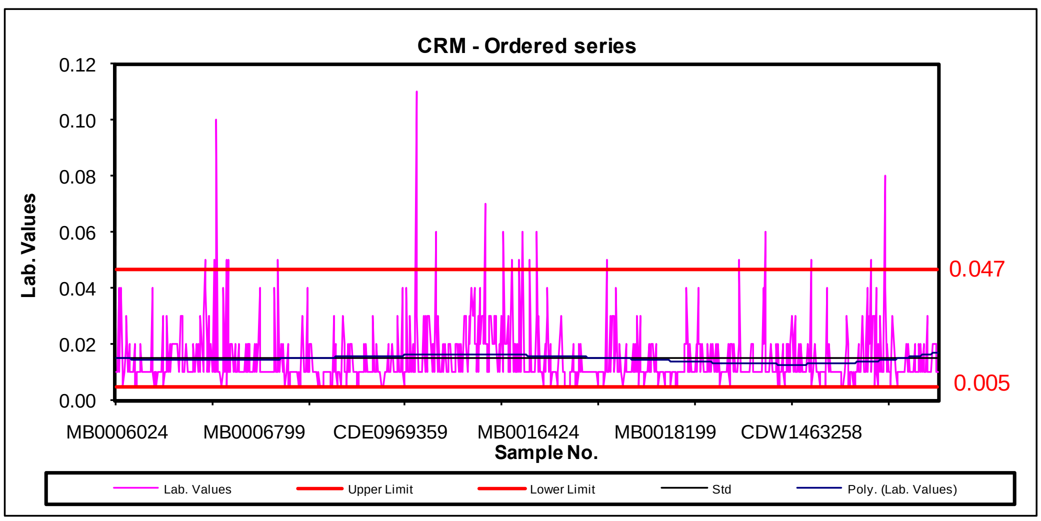

Figure 4-8: Ordered Series of Ni for Blank sample

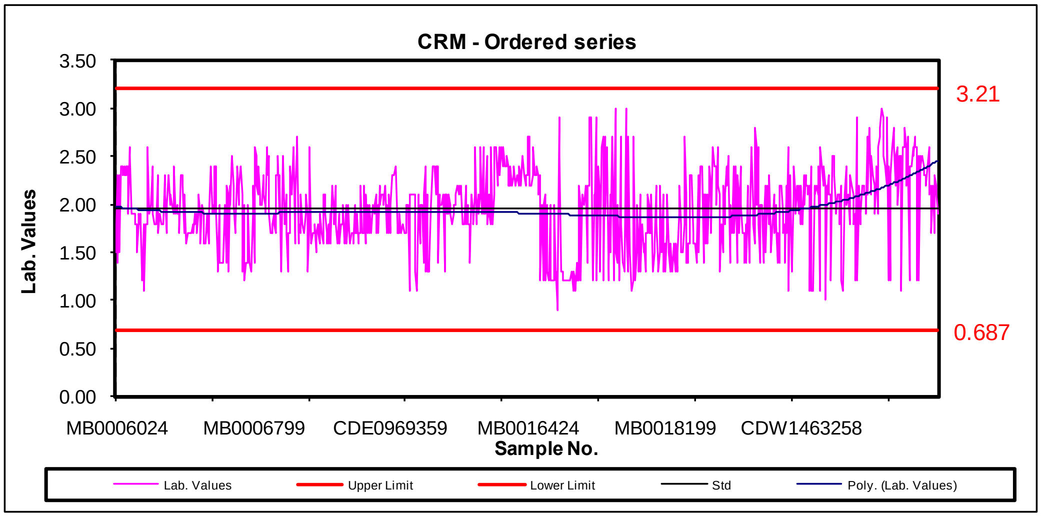

Figure 4-9: Ordered Series of Fe for Blank sample

Duplicates

Pairs of samples prepared and assayed in the same manner provide a measure of the random error of sampling. The total error is the sum of the errors due to splitting the initial duplicate, preparing the sample and assaying the sample.

Inter-laboratory repeats are pulps re-assayed at a second laboratory. These may help to define the inter-laboratory precision and may also identify a bias between the two laboratories.

Golder carried out checks on major elements for duplicate samples submitted since the last external audit (AMEC, 2009), i.e. Ni, Fe SiO_2 and MgO values on samples submitted between 2009 and 2010. Only 2009 pulp re-assay data was provided, therefore, the checks were carried out using this data. The result of the duplicate analysis is summarised in Table 4-6.

In general, all duplicate samples show excellent precision with no obvious bias. All duplicate samples have over 90% of the sample with a HARD value below 10%. As expected, DPP show better precision than other duplicate samples. However, outliers that appear to be sample swaps occurred in most sample preparation stage including pulp re-assay duplicates.

Cross laboratories checks show that there are no significant bias between the PTI laboratory and the Intertek laboratory. However, outliers that appear to be sample swaps were also found in cross laboratories check results.

PTI advised that regular audits are undertaken to avoid sample swaps.

Table 4-6: Result of the analysis for the field duplicate and lab duplicate samples of SPA

Sample ID	HRD%	HARD%	No. Samples	Comment
		DPL		
Ni	-0.01%	0.89%	2546	Excellent precision and accuracy
Fe	-0.06%	0.57%	2546	Excellent precision and accuracy
SiO2	-0.03%	0.79%	2546	Excellent precision and accuracy
MgO	-0.07%	1.13%	2546	Excellent precision and accuracy

Sample ID	HRD%	HARD%	No. Samples	Comment
DPS				
Ni	0.01%	0.76%	3199	Excellent precision and accuracy
Fe	-0.01%	0.49%	3199	Excellent precision and accuracy
SiO2	0.01%	0.55%	3199	Excellent precision and accuracy
MgO	0.01%	0.76%	3199	Excellent precision and accuracy
DPP				
Ni	0.01%	0.81%	3195	Excellent precision and accuracy
Fe	-0.03%	0.51%	3195	Excellent precision and accuracy
SiO2	0.01%	0.55%	3195	Excellent precision and accuracy
MgO	0.06%	0.75%	3195	Excellent precision and accuracy
Pulp Re-assay				
Ni	0.04%	0.92%	2095	Excellent precision and accuracy
Fe	0.09%	0.51%	2095	Excellent precision and accuracy
SiO2	-0.04%	0.52%	2095	Excellent precision and accuracy
MgO	-0.09%	0.72%	2095	Excellent precision and accuracy
Cross laboratory check				
Ni	0.08%	1.14%	3410	Excellent precision and accuracy
Fe	-0.38%	1.11%	3410	Excellent precision and accuracy
SiO2	-0.62%	1.15%	3410	Excellent precision and accuracy
MgO	-1.70%	2.49%	3410	Excellent precision and accuracy

Wet Tonnage Factor

Density data was determined by using the HQ drilling core and estimating the volume of the core as being the cross-sectional area times the length. The wet and dry weights of the core samples were then used to respectively determine the *in situ* bulk density generally referred to as Wet Tonnage Factor, or WTF (including moisture) and the dry bulk density (referred to as Dry Tonnage Factor or DTF in some site documents).

The intrinsic assumptions in this method are:

- That the core does not gain or lose moisture between the time it is drilled and the time it is weighed.
- That the diameter of the core is constant.
- That the cores showing extrusion (recovery greater than 100%) are the product of expansion of the core and not incorporation of lateral material from outside the core inner tube. That is, the mass of recovered material comes from a cut cylinder of 62 mm diameter. This is probably not correct.

PTI recognises that the WTF varies within the laterite profile and from deposit to deposit. Nevertheless, the WTF data from the core is not used in the mineral resource or mineral reserve estimates. Instead, average figures for each of the mining areas (West Block, East Block, Petea), based on historical production data, are used.

Data Coding

Drill hole data is initially assigned a rock type coding in the database based on the field logging and adjusted by the results of the sample analyses. Samples are classified as overburden (OB), limonite (LIM), saprolite (SAP) or bedrock (BRK).

Samples are designated as West Block type or East Block type (most East Block deposits and Petea). In the West Block the mineralisation type is further subdivided as Type 1 (ore contains large corestones of BRK), Type 2 (ore contains medium-sized corestones of BRK), Type 3 (ore contains small corestones of BRK which are easily fractured), based on the logged fracture density and RQD.

Database Systems

Data management at PTI is in transition. A new SQL Server relational database, named dataXPLore, has been developed and is being implemented. The database is accessed by the sample preparation facilities and will link to the LIMS at the ProcTech laboratory. It will store all the information on geological logging, sample preparation and assaying when it is fully operational at the end of 2010.

Data validation and derivation of additional variables seems to be satisfactory.

Heavy reliance on grade control sampling and visual identification of mineralisation-waste seems to work reasonably well in practice,

The sample preparation rooms are well set out, spacious, reasonably clean and equipment is in good condition. Senior staff has a good understanding of the procedures and why they are important. Use of barcoding is good practice and probably reduces errors. Use of regular screening tests to monitor the quality of sample comminution is excellent practice.

The assay laboratory is well set out, clean and equipment is in good condition. Staff have a good understanding of the procedures and why they are important. Use of desiccator cabinets is good practice. Use of barcodes is good practice.

The quality control procedures used for the core preparation and assaying are comprehensive, provide information on the precision of the whole sample preparation process, and adequate for resource estimation.

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

Sampling preparation and assaying at SPA has acceptable errors of precision and no significant bias is observed.

Documentation of the precision of sampling using Field Duplicates and of sample preparation and assaying using Pulp Repeats is carried out by PTI. Procedures are in place to monitor the results by laboratories routinely and there is a high awareness of the need for quality data as the basis for resource estimation.

4.12 Mineral Resource Estimation

Geological Modelling

All geological modelling is carried out using Datamine Studio software. The Anoa South resource block model, created in November 2009 was reviewed by Golder with PTI geologists. The geological modelling process can be summarised as follows:

- Core drill hole data is exported from the master database and loaded into Datamine using the HOLES3D process. This includes basic validation checks such as gaps and overlaps.

- Data is desurveyed. All holes are assumed to be vertical. This is a reasonable assumption for this type of deposit.
- Drill hole coding is checked in 3D and adjusted where necessary.
- The LIDAR topography DTM is used to model the surface topography.
- Triangulated surfaces are created by macro from the flagged samples. The points in the LIDAR topography DTM are registered on the top and bottom of LIM and bottom of SAP surfaces. All four surfaces have the same number of points. This reduces the likelihood of crossovers between the wireframes.
- A bottom of bedrock surface is created to limit the size of the block model.
- The wireframes are filled with 12.5 m by 12.5 m by 1 m blocks. Sub-blocking is not used.
- Where the top of LIM surface crosses the topography surface, a macro is used to truncate the top of LIM surface with topography points. In effect, it is assumed that the limonite zone is eroded in some of the more steeply cut valleys. This is a reasonable assumption for resource modelling, though in reality is likely that there is some limonite formation even on the steeper valley slopes.
- The distribution of mineralisation types is interpreted in plan view (2D) using GIS software. Polygons defining the limit of each mineralisation type are imported to Datamine Studio and are used to flag the resource block model. Note that mineralisation types are not treated separately for grade estimation.
- Global average WTF and DKP factors are assigned to the model. In this case, since Anoa South is a West Block type deposit, the WTF is assigned as 1.90 and the DKP/ROM factor is set to 0.25, 0.25 and 0.27 for west type 1, west type 2 and west type 3, respectively.

Golder has reviewed the geology modelling for Anoa South, Petea B and Songko. These deposits are mostly drilled out on a regular 50 m by 50 m drill pattern with vertical holes. In some areas there are large variations in saprolite thickness as shown in the example from Anoa South in Figure 4-10. The thickness of saprolite can vary from less than 1 m to more than 25 m between adjacent holes. This is due to the sometimes irregular nature of the top of bedrock surface and the presence of bedrock boulders within the saprolite zone.

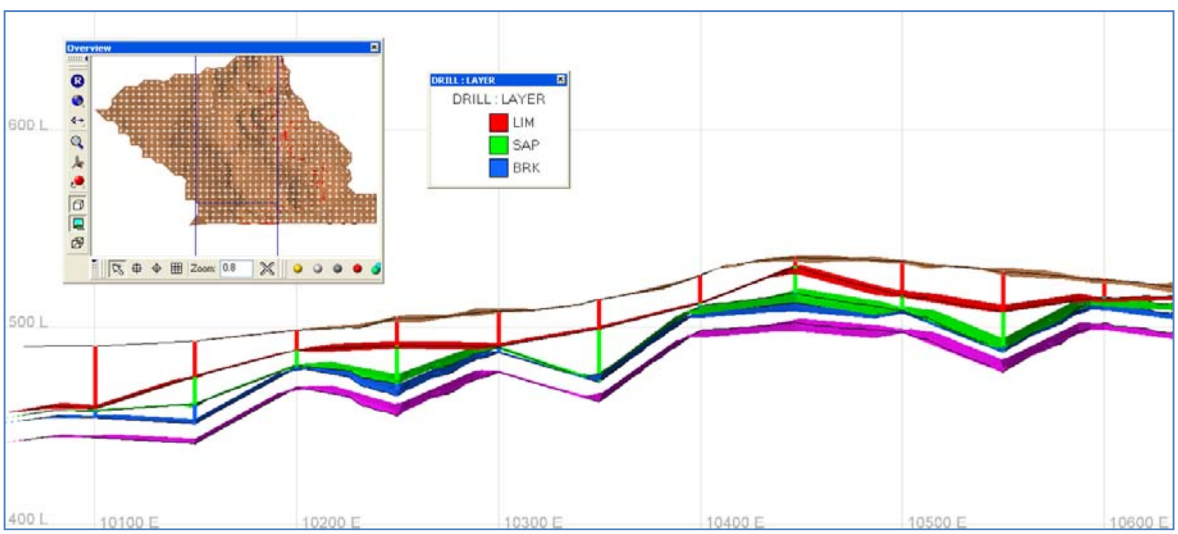

Figure 4-10: Anoa South cross section 7,375 N showing large variations in saprolite thickness

Modelling Areas in the Sorowako Project

The PTI Sorowako Project Area comprises three main regions, West Block, East Block and Petea. The mine operates on a number of small surface mines or 'hills' over several active mining areas in the Sorowako Project. Mining from a series of hills provides an opportunity to blend the run-of-mine (ROM) mineralisation to meet quality specifications of feed for the nickel processing plant. The overall operation comprises numerous individual block models representing the individual hills.

For the purposes of this audit of mineral reserves, Golder selected three areas, as highlighted in Table 4-7, to verify resource estimation procedures and results, including independent validations of the supplied block models.

Table 4-7: Block Model Areas Selected for Audit of Resource Models

Domain	Hill
East Block	Nayoko
	Songko
West Block	Anoa South
	Nickel Hill
Petea Block	Petea B North

Raw Data Preparation, Coding and Compositing

The exploration database contains assay information samples on 1 m intervals. All raw drill hole intervals are pre-coded relative to their position in the lateritic profile. This coding is used to define the following surfaces which form the domains for data analysis and estimation:

- Top of Limonite
- Bottom of Limonite (Top of Saprolite)
- Bottom of Saprolite (Top of Bedrock)
- Bottom of Bedrock

The following grades are estimated through the resource modelling process: Ni, Co, Fe, SiO_2, MgO, Cr, Al, Mn, Ca, and H_2O. The composite data reflects two size fractions, for the -1" mesh and -6" mesh. Both variables are modelled for grade estimation purposes.

Drill hole data was composited to 1 m lengths downhole, compositing within each geological layer (splitting the composites at each boundary).

The treatment of the residual (<1 m) composites at the base of each layer varies between modelling region. For some models, the residual values are retained. For Anoa South, if these short samples were less than 40% of the average sample length (<0.4 m) they were rejected.

For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used) and retaining such short samples will introduce a bias unless length-weighting techniques are used in the grade estimation process.

Sample weighting is used for the saprolite layer. The elements for each size fraction are multiplied by a variable LDW#, defined by dividing the size fraction dry weight by the sample length. This has the effect of ensuring that the size fraction weight is accounted for when a particular size fraction variable is assessed statistically or during grade estimation. For limonite, the original assay values are used without any weighting because almost the entire sample is less than 1" in particle size.

Using the supplied 1 m composite file, Golder examined the distribution of composite lengths for Anoa South using cumulative probability plots, with the distribution for each geological layer shown as overlays, as in Figure 4-11. The proportion of residual composites is generally low, around 0.5% of the global distribution for limonite and around 3% for saprolite. The bedrock domain indicates around 10% of residual composites, although this domain is not of economic importance.

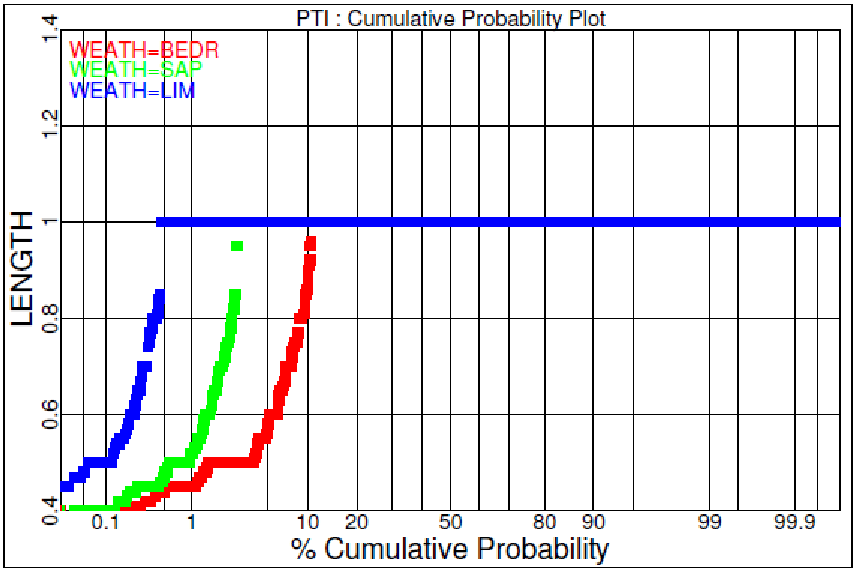

Figure 4-11: Anoa South - Distribution of 1 m composite lengths showing the proportion of residual (<1m) samples

Domains

The geological modelling was used as the input for creating domains for estimation purposes. Interpolation of the various assay elements was done separately for:

- Limonite (generally > 35% Fe and < 5% MgO)
- Saprolite (generally < 35% Fe and > 5% MgO)
- Bedrock

No specific zones or domains were used in the construction of the resource block model other than the vertical layering of the weathering profile.

Exploratory Data Analysis

The PTI report makes reference to the use of statistics in routine validations of the input data. Exploratory Data Analysis (EDA) is presented in the post-2007 mineral resource Technical Documents. The general approach to EDA is fairly consistent between model areas.

Each geological layer is treated separately. This analysis provided a general summary of the distribution characteristics and enabled detection of spurious or outlier values in the data set that may impact on grade estimation.

High-Grade Cutting

The cutting of high-grade samples is often used to control the extrapolation and influence of outliers during the estimation process. PTI carried out checks for misclassification of samples with respect to the lateritic profile, using histograms of all elements and scatter plots of most element pairs. Any apparent outliers were identified and then reviewed. The apparent outliers are taken to represent seams of limonite material within saprolite or boulders of saprolite within limonite. None of the outliers examined were found to be due to incorrect analysis and are all considered valid accordingly high-grade cutting was not implemented.

Variography

Review of Methodology

PTI completed variography studies for deposits in the West Block, East Block and Petea regions, as part of each mineral resource modelling process. The limonite and saprolite domains were assessed. Variography was based on 1 m composites and was undertaken for the Ni, Co, Fe, MgO and SiO_2, Cr, Al, Mn, Ca, H2O, and LDW, for both the -1" and -6" fractions The variograms for the -6" fractions were used to derive parameters for grade estimation by Ordinary Kriging.

The variography procedures were reviewed based on the PTI Report and supported by Golder checks of the variography (see below). The level of documentation of the variography provides some basic information on the approach, but a detailed description of the analysis is not provided.

A summary of the variography procedures is as follows:

- Relative variograms were calculated in an unfolded coordinate system using Datamine software. The unfolding method is based on a vertical projection to a horizontal plane. In the case of the limonite domain, the data was unfolded to the base of limonite wireframe surface. As a consequence, the bottom of the drill hole intercepts all have similar 'elevations' in the unfolded space but the top of the drill hole intercepts have varying elevations according to the length of the intercept. This approach assumes that variations in limonite thickness are primarily due to erosion. Although this is not entirely true, most of the limonite material of economic interest occurs at the bottom of the limonite zone, which is where the unfolding approach will be the most effective. The saprolite domain was unfolded to both the base of the limonite and the base of the saprolite. A proportional method was used, such that the data points were assigned coordinates relative to the parallel upper and lower unfolding surfaces. This approach assumes that each drill hole intercept represents a complete saprolite profile but of varying thickness.

- The relative variograms were modelled using a nugget and three nested structures using a combination of spherical, exponential and Gaussian structures. The nugget variogram was modelled from a down hole variogram calculated in the vertical direction using a 2 m lag. The third variogram structure was used to address any zonal anisotropy between the vertical and horizontal directions, indicated by a difference in the overall sill value between the vertical and horizontal experimental variograms.

- No separation by mineralisation type was made for variography, with all samples from a given model area assumed to represent a particular mineralisation type.

- Saprolite variograms represent the accumulation of LDW# * Grade# for a given size fraction, eg LDW6*NI6. Variograms are also produced for LDW#.

- Directional variograms within the unfolded 'horizontal' plane were calculated. These did not show any clear anisotropy, Hence, isotropic variograms were generated in the horizontal unfolded space. Well-structured downhole variograms were obtained in the minor axis direction.

Block Model Parameters

The model used to constrain the mineral resource estimate is an orthogonal regular block model constructed in Datamine Studio 2 software based on a block size of 12.5 m (X) by 12.5 m (Y) by 1 m (Z). The block model cell represents approximately a quarter of the drill hole spacing in the X and Y direction used for Measured Resources (50 m by 50 m).

PTI advised that most of the block models in SPA use block size on 12.5 m (X) by 12.5 m (Y) by 1 m (Z) as default to accommodate current mine operation practices. In other areas such as SOA and SCD the block size is corresponding to the existing drill spacing.

The vertical block size of 1 m is compatible with the use of 1 m composites. There is no specific reasoning in the report regarding the use of a 1 m vertical block size other than the mineralisation shows significant short-range variation in the vertical direction.

The model provided contained grade variables for -1” and -6” size fractions. Estimation performance parameters such as number of samples and the estimation pass number are also stored in the model.

Grade Estimation

Grade estimation of Ni, Co, Fe, MgO and SiO_2 from 1 m composites was undertaken using Ordinary Kriging in Datamine Studio 2 software. The estimation is constrained to the various geological layers. Each domain was estimated individually using the 1 m composites flagged inside that domain. The estimation was implemented using hard boundary conditions for each of the estimation domains. Golder has previously commented that the use of hard boundaries is appropriate for estimation purposes.

Estimation of the grade in the saprolite layer involved estimating the accumulation of LDW# * Grade# for a given size fraction, eg LDW6*NI6, then dividing the accumulation estimate by the LDW# estimate to achieve final block grades for a given size fraction. This approach was not used for limonite or bedrock, which employed direct estimation of size fraction grades with no weighting.

Search distances and anisotropies were determined from variography. No quantitative neighbourhood testing has been undertaken and a generalised search of 80 by 80 by 5 m was selected as the first estimation pass (a 4 m initial Z search was used for Songko and Petea B North). For a 50 m square drilling grid, the 80 m search is large enough to select the nearest 8 drill holes around a block centroid.

The estimation was performed in three passes. The search distance was progressively increased for each pass using a progressive multiplication factor of 2 for the second pass and 1.5 for the third pass, to enable estimation of more blocks into the block model. Other considerations were the number of samples used and the type of search adopted. Implementation of passes for each layer and for each model area was the same and systematic, with an octant search used except for a normal search on the third pass. No outlier or high grade cutting treatment was used. Table 4-8 provides a summary of the search parameters.

For the Anoa South model, PTI states in their report that 50 % of the blocks were estimated in the first pass, 16 % in the second pass and 34 % pass in the third. The first pass estimation contribution is rather low, at 50%. The minimum number of octants to be filled for the first and second pass is set to 5. This is a very stringent estimation condition and results in several areas of the model not fulfilling this criterion, particularly at the edges and upper and lower vertical limits of the geological layers.

Table 4-8: Generalised Search parameters used for Grade Estimation

Parameter	Pass 1	Pass 2	Pass 3
X-Radius	80 m	160 m	240 m
Y-Radius	80 m	160 m	240 m
Z-Radius	5* m	10 m	15 m
Rotation angle for search volume	0°	0°	0°
Octant Search	Yes	Yes	No
Rotation angle around X	0°	0°	0°
Minimum # of octants	5	5	1
Minimum # of samples in an octant	1	1	1
Maximum # of samples in an octant	4	4	4
Block Discretisation (X / Y / Z)	3	3	1
First Search			
Minimum of total # of samples	20		
Maximum of total # of samples	32		
Second Search			
Multiplying search factor for 2nd search		2	
Minimum # of samples for 2nd search		20	
Maximum# of samples for 2nd search		32	
Third search			
Multiplying search factor for 3rd search			1.5
Minimum # of samples for 3rd search			1
Maximum# of samples for 3rd search			32

* the Z searches for pass 1 to 3 for Songko and Petea B North are 4 m, 8m and 20 m

Volume Variance Correction

PTI used a global change of support process to correct the global distribution of Ni block grade estimates to better match the theoretical variance expected for the required block support. Golder performed independent checks on the correction factors and the change of support process and concludes that the process has been correctly applied. Although the post-processed tonnage curve is a bit out of alignment for the middle range Ni% cut-offs, it is considered reasonable. The theoretical factors used to adjust the block model are optimistic and may lead to an slightly optimistic outcome for Ni grade and an understatement of tonnage at the 1.5% Ni cut-off grade.

Block Model Validation

The PTI report has some discussion on block model validations applied to each block model. A summary of the validation procedures undertaken by PTI includes:

- Visual comparison of estimated block grades against the drill hole data.
- Statistical comparison of estimated global block model average grades against a nearest neighbour estimate global average grade estimate and also global averages of the declustered drill hole data. Nearest neighbour models were created in order to determine the declustered mean to be used to validate the kriged estimates.

- Swath plot validations in easting, northing and RL directions to assess local grade conformance.
- Smoothing effect and variance correction.
- Block Model Peer Review process undertaken on the final block model.

The model validation procedures as described are comprehensive, and validation results are presented in the various PTI modelling reports. This information is useful to demonstrate whether the model appropriately honours the drill hole data in terms of average grade conformance and an appropriate level of smoothing for the mining selectivity has been assumed.

Resource Classification

Mineral resource classification undertaken by PTI was based on a range of criteria, summarised as:

- Drill hole density data coverage.
- Geological and grade continuity.
- Confidence in the grade estimates.

The Sorowako Project area has been progressively infilled from a drill spacing of 400 m down to 50 m spacing. The current mineral resource classification is based on prior experience of the impact of drill hole spacing on grade variability. This has indicated that estimates based on 100 m drill hole spacing showed a significant variation when compared against 50 m spaced drill holes. Accordingly, the drill spacing required to define Measured Resources was set at 50 m.

As the variograms are reasonably defined within the plane of the laterite mineralisation there is enough confidence in lateral continuity, as measured by variography, to classify portions of the model as Indicated or Measured. Indicated Resources are defined based on a drill spacing of 50 m to 150 m, with Inferred Resources defined for a drill spacing of 150 m to 450 m. The mineral resource classification guidelines used by PTI are summarised in Table 4-9.

PTI is planning to supplement the fixed drill density approach to drilling with additional short-scale drilling in areas of irregular terrain and to suit local complexities in the geology. This is a commendable approach to improve confidence in the resource estimates.

Table 4-9: Criteria for Classification of Mineral Resources

	Resource Classification		
	Measured	Indicated	Inferred
Min Sample Density (sites/km^2)	400	399 - 45	44 - 5
Max sample spacing (m)	50	50 - 150	150 - 450

The indicated mineral resource category is used as the basis, with other modifying factors, to assess probable mineral reserves and the measured resource category is used to assess proven mineral reserves.

Mineral Resource Reporting

Golder undertook global volumetric model reports for selected resource block models and found agreement with similar reports contained in some of the resource documentation. However, some areas such as Anoa South quoted the resource in BCM, whilst others such as Nickel Hill and Petea B North quoted the resource in DKP Tonnes. As the model contains no density, mineralisation type variable or DKP recovery factors, Golder is uncertain how these global figures have been derived from the resource block model files.

There are areas of considerable variation in saprolite thickness and areas with very thin saprolite zones. These areas are going to be difficult to mine and reconcile within acceptable limits to the reserve model.

Estimation of the geometry and volume of mineralisation may be higher risk than estimation of grades in the mineral resource models, particularly in wide spaced drilling areas.

Overall, the data preparation, exploratory data analysis, variography, grade estimation and resource classification processes supporting the mineral resources are acceptable.

The lateral parent block size of 12.5 m by 12.5 m is suitable based on the drilling configuration to support Measured Resources, however is too small for wider-spaced regions of some resource areas.

Use of 5 m and 10 m vertical search radii on blocks of 1m height may be excessive. Additional smoothing caused by vertical averaging may distort the final grade/tonnage curve.

4.13 Mineral Reserve Estimation

Golder reviewed the general approach and requirements to defining mineral reserves. Economically viable mineral resources that are used for deriving estimated mineral reserves have to meet the following requirements:

- Minimum cut-off grade constraint established for the current MRMR estimates, based on a proper break-even analysis for the Sorowako project.
- The material has suitable chemistry for feeding to the process plant or after appropriate blending with other mineralisation;
- The material lies within a reasonable distance of the process plant or a scoping study has indicated that no unusual costs are expected over and above the costs of similar material that is currently being mined;
- The material is mineable using current mining methods in place;
- An engineering assessment has been completed of each area confirming the reserve based on extracted pit shells and plans;
- An appropriate allowance has been made for mining dilution and total mineability;
- The area is free of any environmental, forestry or legal encumbrances.

After each Engineering assessment is completed, tonnages are categorised into probable or proven reserves. Based on the following considerations tonnages may be downgraded to a probable level:

- Wide spaced drilling in a remnant mining area;
- Geotechnical issues or concerns that still need to be addressed;
- Areas where mining may impact the local community;

- Environmental work still required;
- Expectation of heavy conglomerate area (i.e. Marlene);
- Low mineability in the pit plans produced.

The overall procedure of applying the economic, geographical, operational and environmental constraints to the mineral resources before they can be considered for the mineral reserves is supported.

Mining

Mining at PTI is by open pit methods using trucks, shovels and excavators. The operation is characterised by moving large quantities of material, mineralisation and overburden and by rehandling the mined mineralisation from the pits, to the screening stations, to the WOS, and to the plant. In addition to mining mineralisation and overburden, competent bedrock is also quarried for sheeting the pit floors. Slag is also used to sheet pit floors.

Mining Method

The mining method is similar to strip mining. The mine is divided into compartments which are mined in a predetermined sequence to meet blending and backfill requirements. The size of the compartments is determined by their backfill volume or capacity.

Mining Models

There are approximately 64 hills covered by the PTI lease, each of which has a reserve/mining model in the 2009 reserve. Each hill is divided into a number of compartments.

The mining models are blocked at 12.5 m x 12.5 m x 1 m. The models have not been regularised, and PTI considers the small block size in the z dimension appropriate to ensure the selectivity required to mine the orebody. The contacts between the overburden, limonite, saprolite and bedrock are irregular as shown in Figure 4-12 and Figure 4-13.

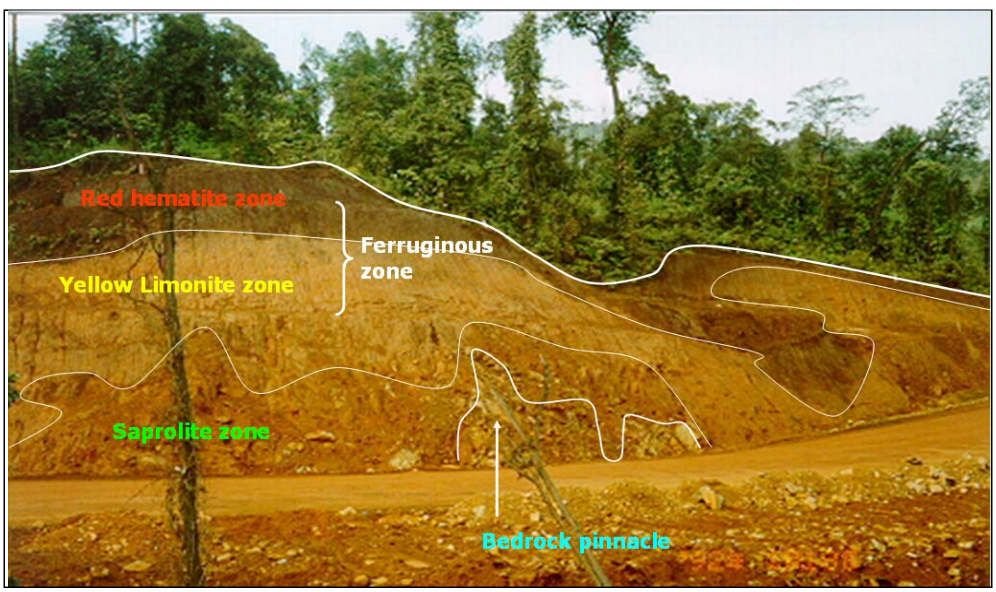

Figure 4-12: Geology Profile West Block (Photo PTI)

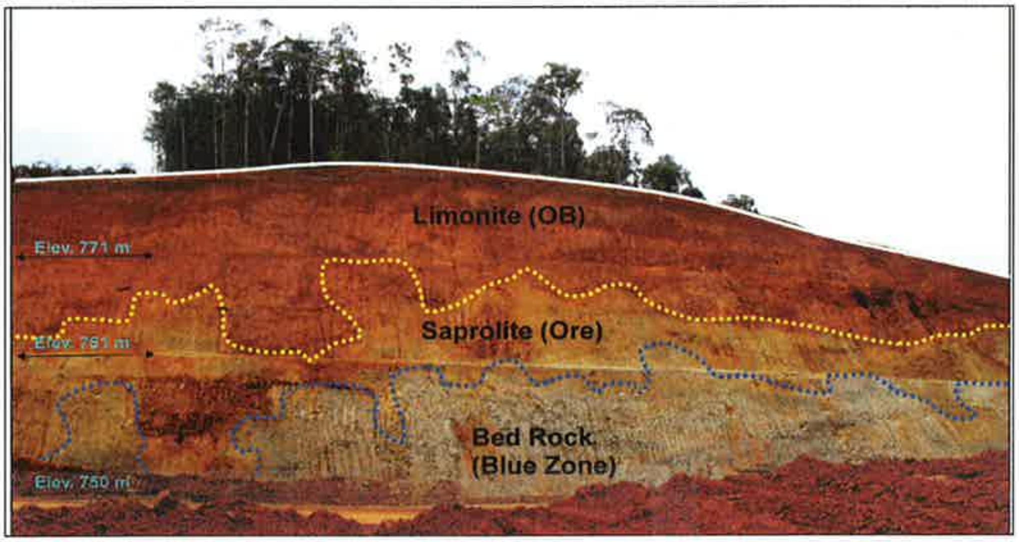

Figure 4-13: Geology Profile Petea (Photo PTI)

PTI makes use of contractors with smaller equipment to mine selectively down to the bedrock where the mineralisation undulates. It is noted that given the drill spacing of 25 m by 25 m or 50 m by 50 m used for resource definition, it is impossible to model all the irregularities between the contacts. Thus mining the irregularities is largely visual and requires suitable equipment to mine the mineralisation.

Once mining has been completed, the resulting pit floor can be very irregular as can be seen in Figure 4-14 and Figure 4-15 which also shows the large boulders which intrude into the saprolite.

Figure 4-14: Irregular Pit Floor (Photo Golder)

Figure 4-15: Irregular Pit Floor Showing Boulders (Photo Golder)

Ore Type Classification

Three mineralisation types are identified at PTI, referred to as West, East and Petea. West mineralisation is further segregated into Types 1, 2 and 3 based on quantity and hardness of the boulders contained within the mineralisation zone.

The three mineralisation types have different characteristics and are blended in a ratio of 68% West to 32% Petea or East. The mineralisation type characteristics are shown in Figure 4-16.

Class. Parameters	WEST BLOCK ORE TYPES			EAST BLOCK & PETEA ORE TYPES			
Main ore types	-1" Type-1	-1" Type-2	-1" Type-3	-1" Hi Olivine	-1" Lo Olivine	-6"	-18"
Ultramafic rock	Generally harzburgite with some dunite			Generally lherzolite at Sorowako East Block; generally harzburgite at Petea.			
Level of serpentinisation	Nil <5%	Low 5-10%	Low 10-15%	Low 10-15%	Medium 15-25%	High 30-60%	V. High 60-100%
Magnetic susceptibility	Very low			Generally low		Very high	
SiO_2/MgO ratio	2.2 – 2.6			1.8 – 2.0	1.6 – 1.8	1.4 – 1.6	
Iron content	High: 20-24%			High: 20-24%		Low: 15-18%	
Optimum screen size (upgrading)	-1"			-1"	-1"	-6"	-18"
Screen recovery ROM to DKP	25-30%			35%	35%	50%	60%
Alternate screen size to improve screen recovery	No alternate size available			No alternate size	-6"	-18"	Entire ROM
Screen recovery using alternate size	No alternate size available			No alternate size	50%	60%	67%
Hardness of boulders	Extremely hard	Very hard		Medium hard	Medium hard	Soft	Soft
Fracture density	Very low	Medium	Medium	Medium	High	Very high	Very high
Difficulty of mining	High	Medium (+)	Medium (-)	Low	Low	Low	Low
Saprolite thickness drilled by auger	<1.5m	1.5 – 4.5 m	>4.5m	Note used for classifying East Block ore types			
Olivine content in the beneficiated ore	Medium	Medium	Medium	Low	Low	Low	Low
Olivine content in the rocky fraction	High			High	Low	Low	Low
OB thickness	Medium			Generally high except at Petea			
Ore thickness	High			Medium			

Figure 4-16: Ore Type Classification (PTI)

The resource models are created in Datamine and converted to Vulcan for the reserve or mining model. A Vulcan script is used for this and performs the following functions:

- Allocates density using mineralisation types and locality;
- Introduces ore loss based on the cgl (conglomerate) field;
- Tonnage estimates for run of mine (ROM), Screening Station Product (SSP) and Dry Kiln Product (DKP);
- Recovered nickel, and contained metal in DKP for Ni, Co, Fe, SiO_2 and MgO;
- Cost and revenue per block
- Defines the product type and reserve class based on drill spacing and mineralisation type.

The mining model uses the material types defined in Table 4-10.

Table 4-10: Definition of Mining Material Types

Type	Definition	Comments
bz	Blue zone	Basement – unweathered rock
ore	Mineralised material with Ni grade >1.5%	Generally saprolite, but some limonite included for blending. Includes sub grade material <2 m thick
hpal	High pressure acid leach	Material identified for HPAL should a plant be built
ob	Material with Ni < 1.5%	Topsoil, limonite and saprolite
waste	Waste	Sub grade material between layers of mineralisation >2 m thick. Mineralised grade material <2 m thick is categorised as waste.

The model tonnes are determined by the following formula, which introduces ore loss to specific blocks based on mineralisation types :

tonnes = volume * density * cong_factor,

where the cong_factor is either 0.5 or 1.

The cong_fact is described in the Vulcan script as a dilution factor while it is actually an ore loss factor. The ore loss should be added to the waste.

The mineral reserve tonnage is declared in DKP tonnes. The DKP tonnage is estimated by applying modifying factors to the run of mine (ROM) tonnes and depending on the mineralisation type, it can vary between 25% and 60% of the ROM tonnes.

The modifying factors used to estimate the SSP and DKP tonnes are shown in Table 4-11.

Table 4-11: SSP and DKP Factors

Ore Type	Type	SSP	DKP
West	1	0.52	0.25
	2 & 3	0.56	0.27
	4 (HO)	0.70	0.34
Petea	5 (-6")	0.78	0.50
	6 (-1")	0.70	0.34

The factors are applied against the ROM tonnes individually i.e.

SSP tonnes = ROM tonnes * SSP factor; and

DKP tonnes = ROM tonnes * DKP factor.

Plant recovery factors are applied as shown in Table 4-12.

Table 4-12: Plant Recovery Factors

Ore Type	Plant Recovery	Nickel Factor
West	0.875	0.95
Petea	0.875	0.931

Cut-off Grade

The cut-off grade for the operation is 1.5% Ni. The cut-off grade is not an economic cut-off but is determined by chemistry of the Ni, Fe and the Silica Magnesia (S/M) ratio. Any change to the cut-off grade would require changes in the plant to accommodate the changed feed grade.

The plant feed specification for Fe and the S/M ratio is shown in Table 4-13.

Table 4-13: Chemistry Specifications

	Minimum	Optimal	Maximum
Fe	19.50	20.25	21.00
S/M ratio	2.05	2.10	2.15

Appropriateness of the Mining Methods

The mining method is open pit mining using truck and excavator methods operating under wet conditions and in material that generally does not require blasting. The pits are located along the sides of hills, and are generally shallow. A typical profile of the orebodies with overburden and mineralisation widths is shown in Figure 4-5. The average overburden height is about 12 metres and the average mineralisation thickness is about 6 metres.

Compartments

Each hill is divided into compartments. All overburden and waste is dumped into mined out pits (compartments). The size of the compartments is a function of backfill capacity. Overburden from a current mining compartment will be used to backfill a previously mined one. The mining sequence of the compartments is aimed at meeting blend and backfill requirements. Figure 4-17 shows a hill split into compartments.

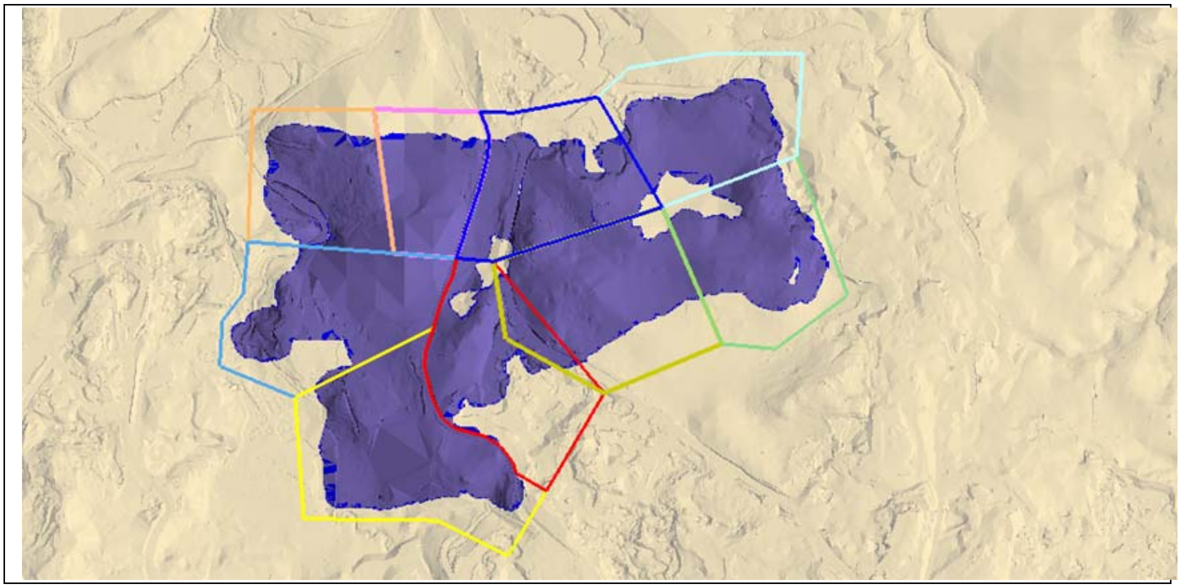

Figure 4-17: Typical Compartments (PTI)

Infrastructure associated with mining hills and compartments includes access roads, cut-off drains and sediment ponds. This is shown in Figure 4-18.

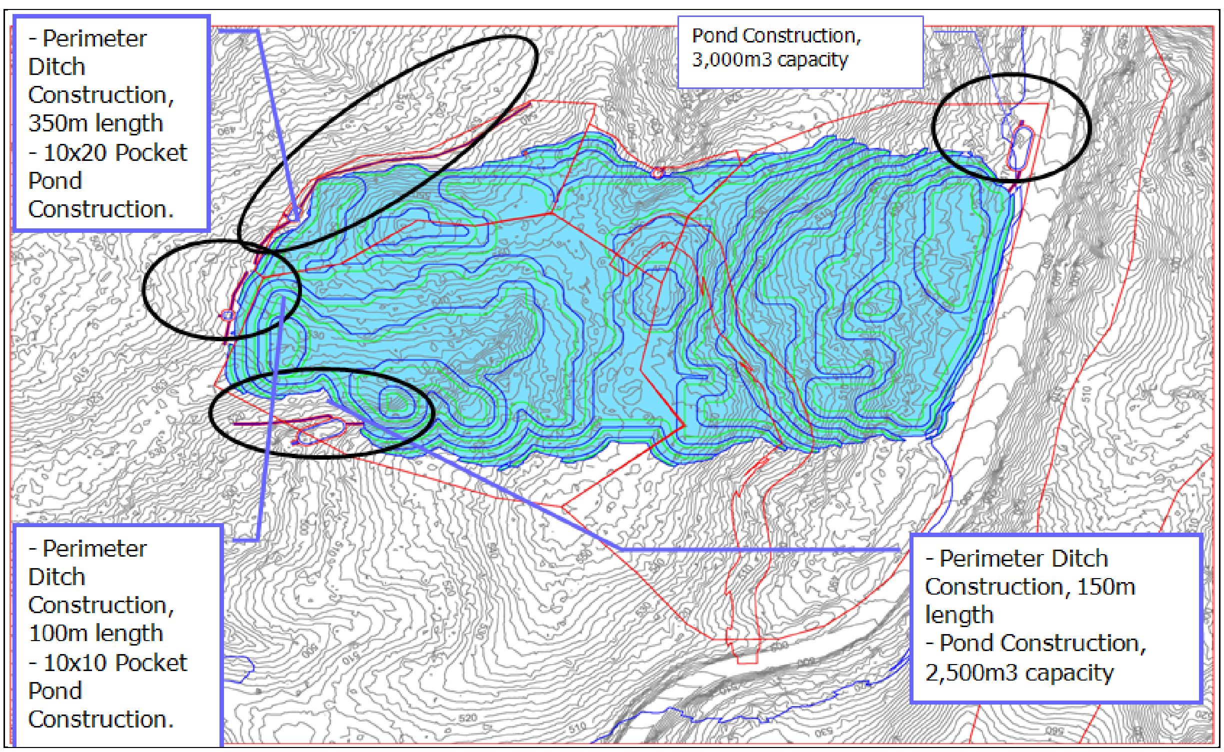

Figure 4-18: Hill and Compartment Infrastructure (PTI)

The total open disturbed area is limited to 1100 ha by lease conditions which therefore restricts the number of compartments that can be open for mining. Management at PTI have set themselves a target to maintain the open disturbed areas at less than 1,000 ha and the planned disturbed and rehabilitated areas for the for the five year plan is shown in Table 4-14.

Table 4-14: Planned Disturbed Area

Year	New Area Disturbed	Area Rehabilitated	Total Disturbed Area
	ha	ha	ha
2008 Actual			988.9
2009 Plan	100.1	104.7	984.3
2010 Budget	117.0	117.0	984.3
2011	126.4	132.0	978.6
2012	100.1	127.1	951.6
2013	126.5	141.4	936.6
2014	99.2	120.1	915.8

The compartment designs appear to overlap one another, and it seems that the different compartments have not been linked back to a final pit design. This is shown in Figure 4-19, which shows the overlap of the different compartments on the Anoa South 500 bench.

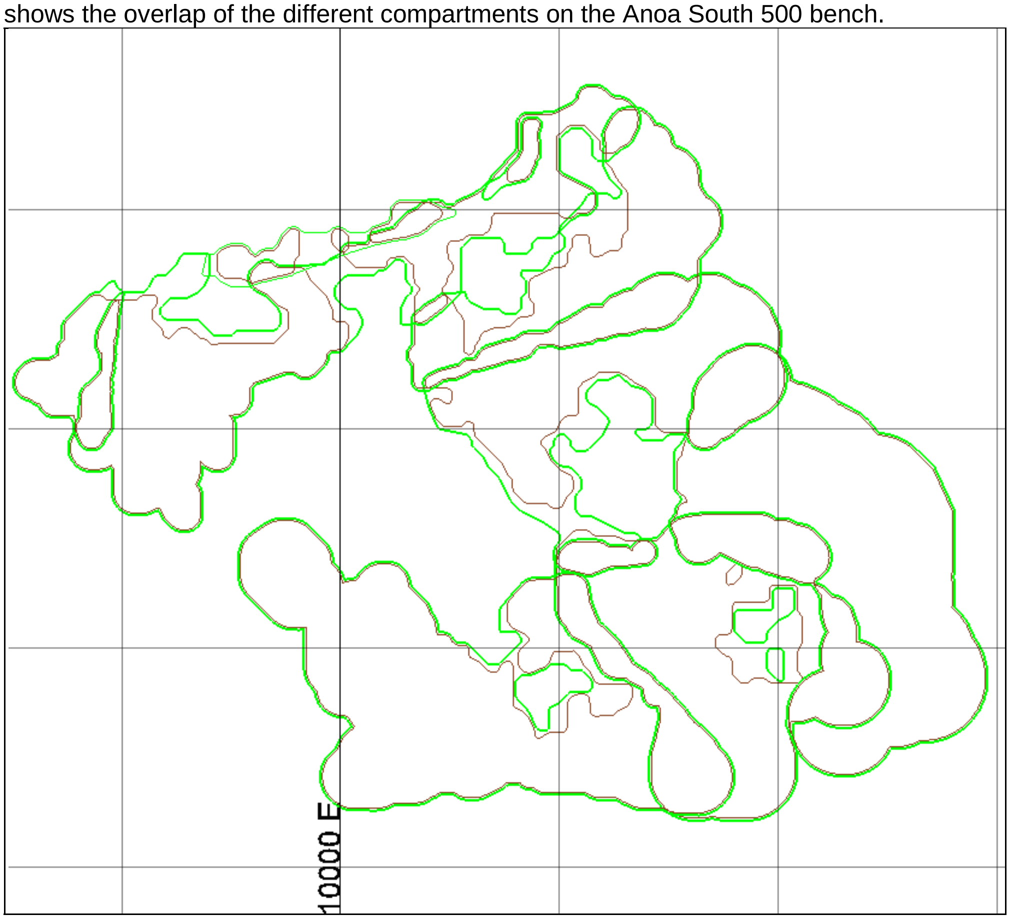

Figure 4-19: Anoa South 500 Bench

Trafficability

Maintaining access into the pits during all weather conditions is essential to maintaining production rates. The material mined in the pits is soft and generally wet, with up to 35% moisture. The floors have to be sheeted to enable trucks and other vehicles to drive on them.

The sheeting consists of between one and one and half metres of either quarried rock, reject rock from the screens or slag. The slag is used on the overburden parts of the floor and the reject and quarried rock is used on mineralisation (Figure 4-20). This material is reclaimed for reuse once mining on a bench has been completed. Sheeting is also required on the disposal areas and up to two metres of quarried material is needed to maintain the trafficability in the disposal areas.

The measures taken to ensure trafficability in the pits and on the disposal areas appear to be successful.

Figure 4-20: Use of Waste Material and Slag on Pit Floor (Photo PTI)

Ore and Waste Mining

Mining of the compartments is carried out in a number of steps:

- Topsoil Removal;
- Overburden;
- Ore; and
- Bottom Ore.

Topsoil Removal

The hills are heavily forested, and the trees have to be knocked down. The topsoil and trees are then pushed up into heaps using small bull dozers (Figure 4-21). The material is then loaded onto trucks and transported to a topsoil stockpile. Rehabilitation is integral part of the mining operation and the topsoil is placed on rehabilitated areas as soon as possible. Given the high rainfall at PTI it is difficult to comply with some of the accepted guidelines on topsoil stockpiles. General guidelines include:

- Construction of stockpiles to minimise deterioration of seed, nutrients and soil biota due by avoiding topsoil collection when saturated following rainfall as this promotes composting;
- Minimise the duration of stockpiling, periods longer than about six to 12 months may cause structural degradation and death of seeds and micro-organisms, especially when soil moisture content is high.

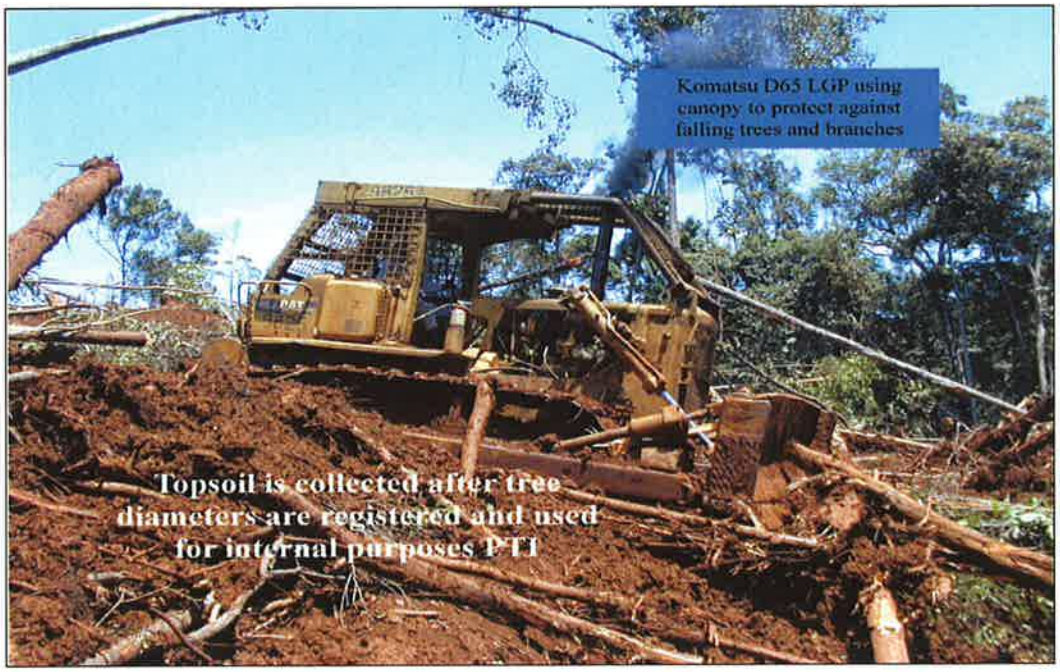

Figure 4-21: Topsoil Removal (Photo PTI)

Overburden Removal

Over 27 million tonnes of overburden are planned to be mined in 2010.

Overburden is mined by a face shovel. The bench height is generally ten metres, but if greater than this, bulldozers are used to push overburden towards the shovel. Where the contact between overburden and mineralisation undulates, bulldozers are used to remove overburden and expose mineralisation in the face ahead of the shovel (Figure 4-22). The aim of this is to enable selective mining and reduce dilution.

Figure 4-22: Overburden Stripping to Expose Ore (Photo PTI)

The shovels are equipped with 12 m^3 buckets and are used for bulk mining of overburden, but may also be used to mine mineralisation if the mineralisation thickness justifies it. The shovels load 100 t trucks, but as observed during the site visit, 50 t trucks are also sent to the shovel. It is not known what implications this mismatch between truck and loader has, and it has not been investigated by site personnel.

PTI subsequently advised that by design, the loading of 50 ton truck is by backhoe, however at the time of the audit site visit, the backhoe was out of commission and temporarily the 50 ton trucks were sent to the shovel.

Mining

Mining the contacts between mineralisation and waste requires selective mining, and for this reason mineralisation is mined using bulldozers and excavators. The complexity around the contacts has been demonstrated in Figure 4-12, Figure 4-13, and also in Figure 4-23 where the different material types can be identified visually.

Figure 4-23: Ore Identification (Photo PTI)

Excavators are able to free dig bench heights less than 10 metres. The excavators are fitted with 4 m^3 buckets which allows for selective mining. The use of bulldozers pushing towards the excavators has been found to improve the production rate.

Ore mining is supervised by a grade control geologist who directs the excavator and takes samples. The grade control system and recommendations are discussed in Section 4.11.

A top of mineralisation clean-up programme was introduced in 2003. Local contractors scavenge the top of mineralisation with small equipment where the bulldozers are unable to mine as selectively as required. This project has proven successful and PTI purchased small backhoes in 2004 for top of mineralisation cleanup.

A “bottom ore recovery” project was started in 2003 with local contractors using small equipment to mine areas along the bottom contact with the blue zone where the larger equipment couldn’t get efficient access. This is shown in Figure 4-24. This is now standard practise using both contractor and PTI crews. Grade control geologists supervise the mining of the “bottom contact” mineralisation.