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Asian Mineral Resources Limited
NI 43-101 Technical Report

Ban Phuc Nickel Project

Son La Province, S.R. Vietnam

By

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1 Executive Summary

This Summary section should be read in conjunction with the total report in order to understand all the necessary and relevant technical and commercial information.

1.1 Property Description and Ownership

Asian Mineral Resources Limited (AMR) has an approved mining licence located over the Ban Phuc Deposit 160 km west of Hanoi in Son La Province, in the northwest of the Socialist Republic of Vietnam (Figure 1).

Figure 1. Location of the Ban Phuc Project

AMR has been active in Vietnam since 1993 and through its 90% owned Vietnam subsidiary Ban Phuc Nickel Mines Limited Liability Company (BPNM, BPNM LLC) is developing the Ban Phuc nickel sulphide deposit in Son La Province. Construction of the project, which is targeting the production of 70,000 tpa of concentrate containing nickel, copper and cobalt, was suspended in October 2008 due to various factors, including the global financial crisis, the imposition of a 20% export tariff on nickel concentrate, the damage caused by Typhoon Hagupit, and depressed metal prices At the time when the project was placed on care and maintenance over 1,000m of underground development had been completed, including 230m raise-bored ore pass and 80% of all plant equipment had been purchased. Full-scale project construction recommenced in May 2012, following AMR securing funding from a new investor.

Ban Phuc Nickel Mines (BPNM) is an incorporated joint venture company which is owned by:

- AMR Nickel Limited, a wholly owned subsidiary of Asian Mineral Resources (AMR) 90%, and
- Son La Mechanical Engineering Joint Stock Company (Coxama) 10%.

1.2 Tenure

BPNM was granted a seven hectare Mining License covering the Ban Phuc deposit on 17 December 2007.

As prescribed under Vietnam's constitution and Vietnamese Law, all land is owned by the State. Under the Land Law 2003 and the Decrees on implementation of the Land Law 2003, the State allocates land use rights to land users; the land use rights are regulated under the Land Law and its implementing regulations, and managed by the provincial People's Committee (PC) and the provincial Department of Natural Resources and Environment (DONRE) of the province where the land is located.

Land use negotiations are conducted with villagers and government agencies mindful of previous traditional ties to land. Under the Vietnamese Law on Land, the Government (through the provincial PC) will acquire the land required for the Project and lease it to BPNM. Existing land occupiers with certificates of land use right or other proof of occupation will be compensated by the Government; BPNM will reimburse the Government for the purchase costs.

1,072,972.5 m^2 of land has already been acquired by the Government and leased to BPNM for the following purposes:

- 1,000,743 m^2 for the mining area (i.e. 7 hectares for the ML covering the massive sulphide orebody) and construction of industrial auxiliary works, which was leased to BPNM in March 2009;
- 67,567 m^2 for office and camp site, which was leased to BPNM in July 2004;
- 1,197 m^2 for construction of domestic water pipeline and auxiliary facilities, which was leased to BPNM in April 2012; and
- 3,465.5 m^2 for construction of water station, pipeline and auxiliary facilities, which was leased to BPNM in April 2012.

Furthermore, BPNM is in the process of leasing 71,794.5 m2 for containing waste rock and soil from the mining activities of the company.

1.3 Accessibility, Climate, Local Resources and Physiography

The Ban Phuc site is easily accessible. The junction for the provincial highway to Son La is within 35 km by paved road and has been recently upgraded. From Son La, access to Hanoi and the port at Hai Phong is by national highway. Alternate light vehicle access via Son Tay, Thanh Son, Phu Yen, Bac Yen and Ta Khoa provides a shorter travelling time from Hanoi on fair-to-good paved road.

The region essentially has two seasons: a dry season (winter) and wet (summer) season. Winter is cool and lasts from October to March with persistent drizzling rain occurring during February and March. Hot monsoonal summers occur between April and September with occasional typhoon events, generally towards the end of the season.

The Ban Phuc deposit is located within rugged terrain of the mountainous areas in the north-west of Vietnam. The steep-sided Da River Valley traverses the region in a general south-easterly direction. On the northern side, steep mountainous country rises to about 1,200 m near Hong Ngai. On the south side of the Da River similar mountainous country rises to 1,520m.

1.4 Exploration, Geology and Mineralization

The Ban Phuc Nickel Project area has a relatively long exploration history. The methods include drilling, trenching, cross-cutting, underground drives and channels. There is no record of previous production at Ban Phuc.

Initial work in the Ta Khoa region by Vietnamese and Chinese geologists focused on areas of known copper mineralization: Van Sai in 1959-61; Na Lui 1959-60; Ban Bo 1959-60; Na Ka in 1960-62; and Ban Phuc 1959-63. Follow-up reconnaissance work in 1961-1964 delineated several new zones of nickel, with or without copper, in nine areas and copper without nickel in an additional five areas.

The Ban Phuc ultramafic intrusion is one of the larger of such bodies in the district outcropping over an area of roughly a quarter of a square kilometre. The Ban Phuc ultramafic is exposed in a window comprising a basement metamorphic complex of Devonian age metasedimentary and metavolcanic rocks. The ultramafic-mafic intrusives are considered to be Triassic in age, although some are postulated to be lower to mid-Palaeozoic.

A number of types of mineralization are recognized in the host intrusive and surrounding metamorphic rocks:

- Massive sulphide type mineralization (MSV);
- Disseminated sulphide type mineralization surrounding the MSV;
- Low grade disseminated sulphides in dunite (DISS); and
- Oxidized type mineralization.

Most nickel mineralization, with or without copper, is both spatially and temporally associated with ultramafic intrusions including:

- veins of high-grade massive nickel-copper sulphide in metasedimentary wall rocks adjacent to ultramafic intrusions, with locally developed low-grade disseminated copper-nickel mineralization marginal to the MSV; and
- disseminated low-grade nickel or nickel-copper mineralization (DISS) in basin shaped cumulate layers (locally multiple), often near the base and walls of ultramafic intrusions.

The concession area lies in the Song Da rift, a major crustal suture zone, which is part of a broader northwest trending corridor of deep continental rifting known as the Red River Fault Zone. The area is an excellent geological address in a geotectonic and structural zone that has many favourable factors for development of Ni-Cu deposit types such as Norilsk (Russia) and Jinchuan (China). Evidence for magmatism on a regional scale adds to this picture.

Considerable potential exists in the district for large-tonnage, lower-grade deposits of disseminated sulphides within ultramafic intrusions, similar to the DISS style mineralisation. Regional exploration in the Ta Khoa corridor has identified an extensive system of mafic-ultramafic intrusives, a remarkable number of which have associated Ni-Cu massive or disseminated sulphide mineralization.

1.5 Mineral Resource

The mineralization models used for the current Mineral Resource estimate are based on interpretations generated by AMR geologists and compiled by CSA Global Pty Ltd (CSA). The mineralization comprises of predominantly steep-dipping lodes to the northeast within MSV and ultramafic rocks. The Mineral Resources estimate for the deposit was completed using Ordinary Kriging (OK) interpolation methods and is reported in Table 1 and Table 2 below.

Table 1 Ban Phuc MSV Mineral Resource Estimate - Summary of in-situ tonnes and grades

Ban Phuc MSV Grade Tonnage Reported above a Cut off of 0.40% Nickel as of 7th September 2012										
Category	**Tonnes (MnT)**	**Ni Grade (%)**	**Cu Grade (%)**	**Co Grade (%)**	**S Grade (%)**	**MgO Grade (%)**	**Fe Grade (%)**	**Nickel (000'T)**	**Copper (000'T)**	**Cobalt (000'T)**
Measured	0.73	2.78	1.16	0.07	13.53	4.39	26.09	20	8	1
Indicated	0.96	2.60	1.22	0.06	12.94	2.04	25.01	25	12	1
Measured + Indicated	**1.69**	**2.68**	**1.19**	**0.06**	**13.20**	**3.06**	**25.48**	**45**	**20**	**1**
Inferred	0.17	1.94	0.80	0.03	10.04	6.76	20.27	3	1	0

** Differences may occur due to rounding errors.*

The resource reported Table 1 is for MSV hosted mineralization occurring below the base of complete oxidation (BOCO) down to the 1100 m RL, which is approximately 250 m – 300 m below surface. DISS Mineral Resource estimate results are presented in Table 2.

Table 2 Mineral Resource estimate results for Ban Phuc DISS Deposit

Ban Phuc DISS Grade Tonnage Reported above a Cut-off of 0.90% Nickel										
Category	**Tonnes (MnT)**	**Ni Grade (%)**	**Cu Grade (%)**	**Co Grade (%)**	**S Grade (%)**	**MgO Grade (%)**	**Fe Grade (%)**	**Nickel (000'T)**	**Copper (000'T)**	**Cobalt (000'T)**
Measured	0.2	1.05	0.15	0.01	1.14	15.83	3.75	2.1	0.3	0.0
Indicated	0.7	1.23	0.14	0.02	0.53	21.69	5.58	8.4	1.0	0.1
Measured + Indicated	**0.9**	**1.19**	**0.14**	**0.02**	**0.67**	**20.37**	**5.17**	**10.5**	**1.3**	**0.1**
Inferred	0.4	1.14	0.04	0.00	0.09	5.93	1.66	4.4	0.2	0.0

** Differences may occur due to rounding errors.*

1.6 Mineral Reserve Estimate

The mine planning for the Massive Sulphide Vein (MSV) zone at the Ban Phuc Nickel Mine (BPNM) in Vietnam was completed by Australian Mine Design and Development Pty Ltd (AMDAD) in 2012 to update the life of mine plan and prepare a Mineral Reserve estimate, including the following:

- Utilise the geological wireframe provided by CSA to produce a minable shape for the MSV wireframe;
- Modify underground development design to suit 2.5m minimum stoping width and planned mining methods;
- Review the June 2010 draft geotechnical report from Pells Sullivan Meynink Pty Ltd (PSM) titled, “Geotechnical Review of Stoping, Ban Phuc”, and implement recommendations into AMDAD’s design work;
- Estimate Mineral Reserves, based on PSM’s geotechnical advice for unfilled up-hole retreat benching, using a 2.5m minimum mining width and 20m sub-level intervals;
- Develop a mining schedule using MineSched software; and
- Estimate mining operating cost, based on the schedule prepared and unit mining costs provided to AMDAD.

The current Mineral Reserve is shown in Table 3.

Table 3 Estimated Mineral Reserve

Item	Mt	Ni grade %	Cu grade %	Co grade %
Proven Mineral Reserve	0.71	2.4	1.0	0.06
Probable Mineral Reserve	0.9	2.1	1.0	0.04
Total Mineral Reserve	1.6	2.2	1.0	0.05

1.7 Mining Methods and Mine Layout

The selected mining method chosen by AMDAD is up-hole retreat benching, without backfill. This method was chosen because of the following factors:

- Simple mining method with lower operating and capital cost method to those involving backfill;
- Top down method, which enables earlier access to ore, and with several benches in operation, the target production rate of 360,000 tpa nominated by BPNM is considered achievable;
- 4.5 m high sill drives are mined along the orebody from the crosscut access on a 20 m (floor-to-floor) vertical spacing;
- These sills are mined at the orebody width although they must be a minimum of 4.0 m wide, to suit the equipment specified, and a maximum of 6.0 m wide. The average orebody width is approximately 3.4 m;
- The crown pillars between vertically adjacent production blocks are set to 7.8 m thick, as recommended by the geotechnical consultants Pells Sullivan Meynink Pty Ltd (PSM); and
- Each stope (including the rib pillar) is set to 20 m along strike. The vertical rib pillars that would be left in each panel have a minimum area (in long-section) of 40 m^2, and would typically be 5.0 m along strike and 8.0 m down-dip. For regions of the orebody above 5.2 m width, the pillar area is set to 1.5 x $(\text{orebody width})^2$.

Access to the underground is via two existing portals.

1.8 Metallurgy

Metallurgical testwork has been undertaken in four phases. Phase 1 and Phase 2 were carried out at Metcon Laboratories in Sydney and Ammtec Ltd in Perth under the direction of Mr. Peter Lewis of Peter J Lewis & Associates. Phase 3 was conducted under the direction of Metplant Engineering Service (Metplant). Phase 4 Testwork was completed at Ammtec under the direction of Mr Steve Ennor of AMR.

Phase 1 Testwork (prior to February 2005) consisted of comminution and preliminary flotation testwork on two composite samples of disseminated mineralization and one composite sample of the MSV. Following a preliminary economic study based on these testwork results the project scope was changed to focus only on underground development of the MSV. Consequently work on the disseminated sulphides was curtailed.

Phase 2 Testwork (March to August 2005) was the definitive testwork on which the current proposals for metallurgical processing are based. Comminution testwork was completed on composite samples of both the MSV and its adjacent waste material taken from the upper, middle and lower parts of the eastern and western sections of the deposit. This testwork showed that there was relatively little variation in the ball and rod mill work indices across the deposit but the waste material was considerably harder than the corresponding MSV, with 20 to 30% higher work indices reported.

Flotation testwork was completed on seventeen composite samples of the MSV that included appropriate amounts of internal and external waste. Six of these represented different locations within the deposit, six a range of nickel head grades from 1.40 to 4.32% Ni, and five a range of Ni:Cu ratios in feed.

The flotation testwork showed that saleable concentrates could be produced with grades of 9% Ni or more at nickel recoveries in the range 85% to 90%. Analyses showed that the concentrates produced had low levels of smelter penalty elements. Algorithms for predicting metallurgical performance were developed from the flotation results obtained on the seventeen composites.

An appropriate amount of supplementary testwork was also completed as required for plant design and concentrate storage and transportation. This included thickening testwork on both final tailings and concentrate, filtration testwork on concentrate, tailings viscosity and consolidation testwork and the determination of the Transportable Moisture Limit for the concentrate and its potential for self-heating.

Phase 3 testwork (2007 to 2008). Flotation testwork was completed on a new sample of MSV and a sample of the disseminated mineralization, both of which were taken from single locations. Very similar flotation performance to that achieved in the Phase 2 testwork was obtained on the new MSV sample. Aging testwork on the sample showed that careful management of the ROM pad stockpiles would be needed to minimize oxidation of the sulphide minerals.

Some encouraging results with respect to nickel concentrate grade and recovery were obtained on the sample of disseminated mineralization. However, the MgO assays of the concentrates produced were well above the levels required for marketable concentrates.

Attempts at selective removal of pyrrhotite from the MSV head sample and MSV concentrate were unsuccessful.

Phase 4 Testwork (April and September, 2011). Flotation testwork was completed on a sample of MSV to investigate the potential for producing separate copper and nickel concentrates. Differential flotation of the nickel and copper was achieved but the grade of the sample tested was well above that expected in practice particularly for nickel.

1.9 Recovery Methods

The Recovery Section 17 is prepared from an assortment of working design documents from Metplant Engineers, understood to be generated in 2008 prior to the cessation of construction and EPCM activity. The previous 2007 Technical Report is based upon throughputs of 250,000tpa and 300,000tpa, with a reasonably detailed process description. The Metplant Process description for the upgraded plant throughput of 450,000tpa is not available. Hence, the information in Section 17 has been developed from the 450,000tpa design criteria document provided by Ban Phuc Nickel Mines, and the assumption that the flowsheet remains unchanged.

The design throughput of 450,000tpa is significantly greater than that required by the scheduled mine ore production to the plant. In the first year of production, 124,000t of ore production is forecast, following which annual ore production increases to approximately 360,000tpa. It is thus clear that the processing plant has significant excess capacity.

The process plant is designed to process up to 450,000 tpa. The process plant will produce a bulk nickel/copper concentrate via a conventional base metals flotation flowsheet, comprising of the following unit processes:

- Crushing;
- Crushed ore storage, reclaim and mill feed;
- Grinding;
- Flotation;
- Concentrate thickening, filtration, storage and load-out;
- Tailing thickening, pumping and return water;
- Reagent storage, mixing and distribution; and
- Utilities.

Process design criteria have been derived from metallurgical testwork, thus allowing generation of flowsheets and equipment lists to develop the process plant design. The process plant location is adjacent to the mine portal for improved haulage efficiency. The process plant layout has been designed to take advantage of the surface gradient in the area.

1.10 Project Infrastructure

The process plant site is on a relatively gently sloped area of West Ban Phuc Valley. A small ROM ore pad is developed at the 230RL level of the underground mine haulage portal and the rest of the site has been developed by cut and fill. Cut-off drains have been developed around the site to divert the valley catchment run-off to downstream of the plant site pads:

- ROM ore stockpile and process plant at 220 RL;
- Concentrate shed at 210 RL;
- Workshop and warehouse at 200 RL; and
- Administration building at 200 RL.

Power is to be supplied from the national 35kV grid power via a 6kV substation for distribution within the site via low voltage motor control centres.

Process water will be recycled from the Tailings Storage Facility (TSF) with make-up and raw water drawn from the Chen Stream which feeds into the Da River. The camp will draw water from the Da River to supply a reverse osmosis plant for domestic (non-potable) water use.

The camp is located 3 km from the mine site on the east bank of the Chen Stream downstream of its confluence with Dam Creek, on a site already acquired by BPNM. The site is 35 m to 100 m wide and approximately 300 m long.

1.11 Market Studies and Contracts

The Ban Phuc Nickel Project will produce a mixed sulphide concentrate containing nickel, copper and cobalt. Nickel, copper and cobalt are exchange traded metals, and the pricing terms of the existing offtake contract are linked to London Metal Exchange prices. As such, no market studies are intended to be undertaken.

BPNM entered into an offtake agreement with Jinchuan Group Ltd on 28 April 2008. Jinchuan agreed to purchase all nickel concentrates produced during the life of the initial Ban Phuc Project. The agreement also granted Jinchuan first refusal option on additional nickel concentrates that BPNM may produce from new projects other than Ban Phuc.

Various other key contracts are described in Section 19.

1.12 Environmental Studies

AustralAsian Resource Consultants Pty Ltd (AustralAsian) completed an Environmental and Social Impact Assessment (ESIA) in September 2005 as part of the Feasibility Studies for the Ban Phuc Nickel Project. The ESIA was subsequently updated by Centre for Environment Consultancy and Protection (CECP) to satisfy the requirements of the Law on Environmental Protection 2005 and other relevant environmental and social-related Vietnamese legislation. The survey covered aspects of meteorology, soil types, local flora and fauna, air quality and noise quality, the findings of the survey are outlined in Section 20.3.

Other environmental clauses relevant for the Ban Phuc Nickel Project, including mine closure requirements; environmental approval; resettlement approvals; and permitting approvals are covered in Section 20.3.

AMR’s engagement with the local communities is guided by four broad principles, as defined by their Corporate Social Responsibility Policy. These four principles are:

- Respect the cultures, customs and values of individuals and groups whose livelihoods may be affected by exploration, mining and processing;
- Recognise local communities as stakeholders and engage with them in an effective process of consultation and communication;
- Contribute to and participate in the social, economic and institutional development of the communities where operations are located and mitigate adverse effects in these communities to the greatest practical extent; and
- Respect the authority of national and regional governments and integrate activities with their development objectives.

A two-phase relocation process was implemented after consultations were conducted in the form of face-to-face meetings at the various villages, at village meetings and at the Muong Khoa Council Office. The relocation was conducted in two phases:

- The Phase 1 resettlement was completed in 2004. A total of 22 households and two organizations were moved to different locations around Son La Province; and

- The Phase 2 Resettlement was completed in 2007 and covers over 100.1 ha of the mine site area. This area incorporates the mining area, plant site, TSF, internal roads, other facilities. A total of 89 households and two organizations were resettled.

Groundwater and surface water sampling has been conducted and assessed against the Vietnamese standards. Overall the water quality was within the Vietnamese standards. Concentrations of heavy metals (Nickel, Iron, Lead, Chromium, Copper, Zinc and Manganese), cyanides and sulphates were also below Vietnamese groundwater standards. Samples of potable water were taken from two typical domestic households in the area and also from the Ban Phuc and Ban Trang water supplies.

The ESIA survey concluded a total of 464 plant species were identified. The project area maintains 5 endemic, rare and valuable species as listed in the Vietnamese Red Book of Endangered Species or in the International Union for the Conservation of Nature and Natural Resources Red List of Threatened Species.

The fauna survey conducted during the ESIA indicated that a total of 13 mammal species from 10 genera and 7 families were reported to occur in the Muong Khoa area, including the Rhesus Monkey (a rare and vulnerable mammal). A total of 31 bird species and of 9 amphibian species were identified within the project area. A further 10 species of reptiles were also identified, with 6 of these being listed in the Vietnamese Red Book as being either threatened or vulnerable. The flora and fauna in the project area currently present no material issues for the project.

The main mine closure and rehabilitation program principles are:

- project area is to be used by local communities for subsistence farming;
- allow for sustainable use of land by the local communities; and
- return the land to its original conditions (consistent with surrounding physical and social environment with no ongoing maintenance required) within a reasonable timeframe.

The full mine closure plan and rehabilitation is discussed in further detail in Section 20.15.

Potential environmental, social and health and safety impacts were identified relating to the construction, operational and closure phases of the project. Mitigations measures have been implemented to reduce the potential impacts that mining activities may have upon the local flora and fauna, groundwater and surface water quality, noise levels and air quality within and adjacent to the project. These Mitigation measures are outlined in Section 20.13.

1.13 Capital and Operating Costs

Estimated capital expenditures for mine equipment costs and construction costs are based on information provided by BPNM.

Capital expenditure from 01 January 2013 to commencement of production at the end of June 2013 totals approximately US$ 34.66 M, inclusive of contingency of US$ 2.80 M (Table 4).

Table 4 Initial Capital Cost Summary

Item	US$
Labour	**6,538,390**
Earthworks	4,419,106
Processing plant	9,446,431
Engineering	1,818,636
Commissioning	1,190,852
Mobile equipment and spares	2,753,602
Camp	308,073
HSE	65,600
Contingency (10%)	2,000,230
Total Construction	**22,002,530**
Mining equipment	500,000
Re-open underground	200,000
Pre-production mining and development	4,621,332
Pre-production mining and development contingency (15%)	798,200
Total Mining	**6,119,532**
Total Capital Cost	**34,660,453**

1.14 Economic Analysis

The development of the project is at an advanced stage, with underground and surface infrastructure near completion and production planned to commence in June 2013.

The Mineral Reserve currently totals 1.61Mt at 2.21% nickel, 1.01% copper and 0.05% cobalt giving a mine life of just over 5 years. After depletion of the MSV Mineral Reserve, the DISS Mineral Resource could offer the potential for a larger, bulk mining operation. Another alternative is to mine a portion of the DISS resource which is readily accessible from the underground and could, subject to metal prices, provide an opportunity to extend the life of the project.

The metal distribution in the upper sections of the ore body lends itself to extraction of high-grade zones early in the life of the project which will facilitate an early payback on capital.

The target mining rate is limited by the size and geometry of the MSV orebody and has been set at 1,000 tpd. The process plant throughput has been designed to match the ball mill which has an annual capacity of 450,000tpa. The mill will be operated in batch mode to meet the target ore production from the mine (360,000tpa). All other aspects of the project have been designed and are being constructed to accommodate the maximum design capacity of 450,000tpa thus creating spare processing capacity should mining opportunities allow.

Annual concentrate production is expected to average around 70,000t and contain 6,400t of nickel, 3,200t of copper and 130t of cobalt. AMR has identified multiple opportunities to grow the project, including:

- Blending of selected higher-grade portions of the disseminated sulphide deposit located adjacent to the MSV and accessible from the planned underground infrastructure;

- Extension of the MSV at depth; and

- Processing of identified satellite deposits which are located within trucking distance from the Ban Phuc processing complex.

The project economic model is based on the mine plan prepared by AMDAD and capital and operating cost estimates prepared by BPNM. The key assumptions are summarized in Table 5. Key Financial Results are presented in Table 6.

Table 5 Key Dates and Economic Assumptions

Key Dates	Units	
Production Commences		30 June 2013
Metal Price		
Nickel	US$/t	19,974
Copper	US$/t	8,333
Cobalt	US$/t	31,724
Production		
Ore mined	t	1,613,413
Ore treated	t	1,613,413
Ore grades (mine life average)		
Nickel	%	2.21
Copper	%	1.01
Cobalt	%	0.05
Process Recoveries		
Nickel	%	85
Copper	%	95
Cobalt	%	70
Concentrate Grades		
Nickel	%	9.50
Copper	%	4.86
Cobalt	%	0.19
Metal produced in concentrate		
Nickel	t	30,310
Copper	t	15,452
Cobalt	t	591
Tax and tariff rates		
Royalty	%	10
Corporate Income Tax Rate	%	25

Export Tariff Nickel and Cobalt	%	20
Export Tariff Copper	%	30
Constants		
Tonnes to Pounds		2,204.62

Table 6 Key Financial Results

Item	Unit	Total
Base Case		
Net Present Value (NPV) @ 12%	US$ (000s)	65,929
Internal Rate of Return (IRR)	%	69

1.15 Conclusions and Recommendations

1.15.1 Mineral Resource

The current resource models provide robust global estimates of the in situ Ni, Cu, Co, S, Fe, Mg mineralization in the Ban Phuc MSV deposit.

Although a relatively small percentage of the total resource, further drilling will need to be carried out if the Inferred Mineral Resource estimate is to be upgraded to the Indicated status. The drilling program design should meet the following guidelines as a minimum:

- At a section spacing of 50 m or less;
- On each 50 m section, two drillholes giving intercepts that cover the full width of the MSV below the base of complete oxidation (BOCO); and
- Additional drillholes should aim to confirm the extrapolated depth extensions of the mineralized lithology. The holes should also aim to delineate the orientation of the interpreted steeply dipping structure. The drillhole intercepts for this purpose should cover mineralization projected below the present Mineral Resource base depth of 1200mRL down to 1100mRL.

1.15.2 Metallurgical Testwork

Marketable flotation concentrates can be produced from the MSV at all locations within the deposit.

1.15.3 Mining Methods

The selected mine plan has been developed in close consultation with the geotechnical consultants to produce a simple mining method, whereby a number of areas of the orebody can be in production at one time, therefore providing a flexible mine to meet the needs of the production profile.

1.15.4 Environmental Studies

Overall, analytical monitoring results for air, water and noise parameters indicated compliance with the Vietnamese legislation. With the exception of occasional non-compliances with dust concentrations detected during January and February 2011, no other exceedances were observed. The occasional historic exceedances are unlikely to significantly impact the operations of the mine.

AMR is currently in the process of developing an internal monitoring plan to help keep track of resettlement activities associated with the Ban Phuc Nickel Project.

1.15.5 Economic Analysis

The project is sufficiently robust to sustain a positive return even at low nickel and copper prices.

CSA recommends that BPNM consider the following:

- Obtain written quotes from original equipment manufacturers for the underground equipment (this equipment is expected to be used for the life of the mine);
- Obtain signed contracts for the transport of concentrate;
- Obtain signed contracts for the Laboratory Contract;
- Submit the Mining Licence Amendment (Table 7) for the new tonnage requirements (the current mining licence refers to ore throughput of 200,000 tpa); and
- Studies should be carried out to assess the options for increasing the life of the mine.

2 Introduction

2.1 Issuer

This report has been prepared at the request of Asian Mineral Resources Limited.

2.2 Sources of Information

Seven consultants have been involved in the preparation of this report. They act as independent qualified persons in their respective disciplines. Each is listed below with their respective items of responsibility and sources of information.

2.2.1 CSA Global Pty Limited (CSA)

Bielin Shi is the independent qualified person for geological and resource estimation aspects of the report. This includes Sections 4, 9, 10, 11, 12, 14, 25.1.1 and 26.1

Gerry Fahey is the independent qualified person for compiling the technical report. This includes Sections 3, 5, 6, 7, 8, 18 and 20. I am jointly responsible for Sections 1, 2, and 25.

Andrew Kinghorn is the independent qualified person for compiling the technical report. This includes sections 21, 22 and 25.2.2 of the Technical Report and jointly responsible for sections 18 and 19.

2.2.2 Australian Mine Design and Development Pty Limited (AMDAD)

John Wyche of Australian Mine Design and Development Pty Limited is the independent qualified person for reserve, mine design and scheduling in the technical report. This includes Sections 15, 16 and 21.2.1.

2.2.3 Peter J Lewis & Associates

Peter Lewis of Peter J Lewis & Associates is the independent qualified person for metallurgical testwork. Peter Lewis planned, supervised and interpreted the Phase 1 and Phase 2 testwork at Metcon Laboratories Pty Ltd in Sydney and Ammtec Ltd in Perth and, in conjunction with AMDAD, selected the composite samples used in this testwork. He was also responsible for the preparation of Section 13, 25.1.2 and 26.2 of the Technical Report and jointly responsible for Sections 1 and 2.

2.2.4 Independent Metallurgical Operations Pty Ltd (IMO)

Tom Gibbons (FAusIMM) of IMO is the independent qualified person for the process plant design. This includes Section 17 and 25.2.1 of the Technical Report and jointly responsible for sections 18.Sources were Ausenco and Metplant Engineers for process design.

2.3 Scope of Personal Inspections

2.3.1 CSA

Bielin Shi and Gerry Fahey visited the project area for four days in June 2010.

Andrew Kinghorn visited the property for three days in October 2012.

2.3.2 Australian Mine Design and Development Pty Limited

John Wyche visited the property for two days in February 2004.

2.3.3 Peter J Lewis & Associates

Peter Lewis visited the property for two days in February 2004.

2.3.4 Independent Metallurgical Operations Pty Ltd (IMO)

Tom Gibbons visited the property for three days in October 2012.

2.4 Terms of Reference

Information detailed within this report has been provided by various technical experts including:

CSA Global, AMDAD, Peter J Lewis & Associates, and Independent Metallurgical Operations Pty Ltd (IMO).

2.5 Sources of Information

All databases and background information was provided by AMR. The following reports were the main contributors to the current document.

- Ban Phuc Nickel Technical Report 2007 by Sandercock et al (2007).
- Mineral Resource Update – Ban Phuc Nickel-Copper Prospect. NI 43-101 Report by Hellman and Schofield Pty Ltd (2004).
- Comminution tests on drill core representing the host rocks in the Ban Phuc copper/nickel deposit, Vietnam: - Report M0775A by Metcon Laboratories (May 2004)
- “Ban Phuc Nickel Project, Vietnam. Flotation Testwork on Disseminated Sulphides. Stage 1”. Report M0755B by Metcon Laboratories (March 2005)
- “Ban Phuc Nickel Project, Vietnam. Flotation Testwork on Disseminated Sulphides. Stage2”. Report M0755C by Metcon Laboratories (December 2005)
- “Ban Phuc Nickel Project, Vietnam. Flotation Testwork on Massive Sulphides. Stage 1”. Report M0755D by Metcon Laboratories (October 2005)

- "Ban Phuc Nickel Project, Vietnam. Flotation Testwork on Massive Sulphides. Stage 2". Report M0755E by Metcon Laboratories (October 2005)
- "Comminution testwork conducted upon samples of ore from the Ban Phuc nickel and copper deposit." - Report No. A9634 by Ammtec Ltd. (April 2005)

The following Microsoft Excel Spreadsheets supplied by AMR to CSA and AMDAD were the main source documents for the information in Sections 15, 16, 21 and 22.

- AMR Technical Report_BPNM Monthly Model_27012013
- Budget Summary for NI 43 101

The following reports supplied by AMR to AMDAD were the main source documents for the hydrogeological and geotechnical information in Sections 16.

- 'PSM1166.R3 DRAFT, *Geotechnical Review of Stoping Ban Phuc Nickel Mine, 23rd June 2010 by* Robert Bertuzzi
- *PE701-00019/6, Ban Phuc Project – Tailings Storage Facility, Water Management and Plant Site Foundation Assessment – Feasibility Study by* Knight Piésold.

3 Reliance on Other Experts

CSA are not qualified to comment on issues related to legal agreements, royalties and permitting matters. The authors have reviewed the mining titles, their status and the technical data supplied by AMR's management and technical information in the public domain. The authors have relied on presentations and documentations supplied by AMR's management.

CSA are relying completely on the expert knowledge, reporting and opinion of Mr Andrew Bedford (FGS, AIEMA) of KBC environmental consultants who conducted an investigation into the environmental, social and community impact on the deposit. CSA have relied completely on the report entitled *'200019, Ban Phuc Nickel Mine, 6th November 2012 by Andrew Bedford'* for Section 20 of this report.

4 Property Description and Location

4.1 Area of Property

The mineralized zones covered by this Technical Report lie within a granted Foreign Investment License 522/GP with an area of 150 km², the Ta Khoa Concession.

4.2 Location of Property

Figure 2 shows the extents of these tenements relative to the mineralized zones currently interpreted for Ban Phuc. This figure shows only the tenements hosting the currently interpreted mineralized zones. BPNM has continuous tenement coverage between the north-western and south-eastern groups of deposits at the Ban Phuc project.

The Ban Phuc tenements are centred on approximately latitude 21° 9’ and longitude 104° 33’ and are located approximately 160 kilometres due west of and 230 km by road from Hanoi, Son La Province, S.R Vietnam.

Mining License boundaries are defined by the location of corner claim pegs with approximate coordinates based on GPS readings recorded in claim documentation.

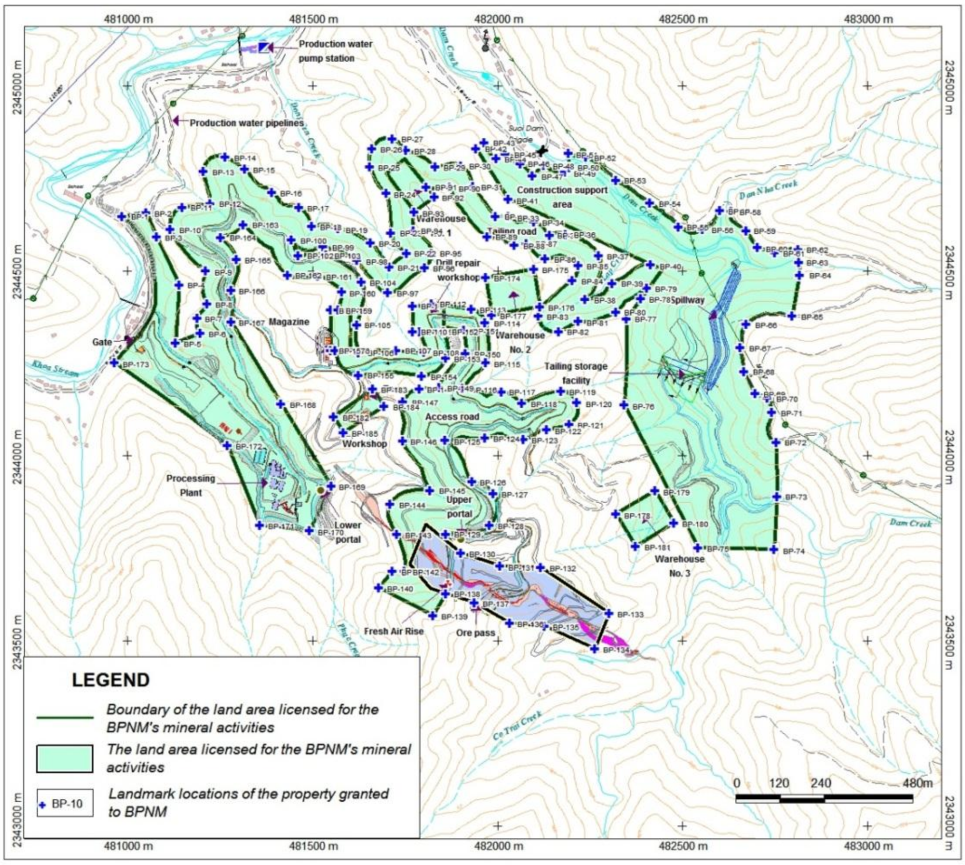


Figure 2 Licenced areas

4.3 Tenure Agreements and Encumbrances

BPNM is an incorporated joint venture company which is owned by:

- AMR Nickel Limited, a wholly owned subsidiary of AMR – 90%, and
- Son La Mechanical Engineering Company Joint Stock Company (Coxama) of Son La Province – 10%.

In January 1993, BPNM was granted Investment License (IL) 522/GP with an initial term of 20 years, giving exclusive rights for exploration and mining. After statutory relinquishments, the concession area has now been reduced to 150 km^2 covering the Ban Phuc deposit and adjacent exploration areas.

BPNM is now operating under the current Investment Certificate No. 241022000033, as amended. Under this Investment Certificate, BPNM is licensed to a project area of 150 km^2. Within the 150 km^2 area, a seven hectare Mining License covering the Ban Phuc deposit was granted to BPNM on 17 December 2007. If BPNM wants to carry out any exploration activities within the 150 km^2, BPNM must obtain exploration licenses.

The Investment Certificate contains provisions related to BPNM's status, capital and scope of business as an enterprise and provisions which are project specific. Within the first category, the Investment Certificate confirms the existence and status of BPNM as a limited liability company headquartered at Muong Khoa commune, Bac Yen District, Son La Province with a charter capital that has been contributed 90% by AMRN and 10% by Coxama and carrying on until 2043, the business of exploring for, mining and processing nickel-copper ore at Ban Phuc mine. Project specific provisions include:

- Project targets which are implementation of business and exploration activities on the Ban Phuc project area and other project activities; implementation of feasibility studies on targeted areas within Ban Phuc project area to exploit copper, nickel, cobalt and associated minerals; development of mines to exploit copper, nickel, cobalt and the associated minerals within the Ban Phuc project area; exploiting, processing, transporting and exporting nickel and copper ore and associated minerals selling all mining products of the Project.

- Project scale: the granted project area of 150 km^2 at Ban Phuc mine area.

- Operation period of 50 years from January 29, 1993.

- Project schedule: BPNM will import machinery and equipment for project development and mine construction and will implement nickel ore mining and processing activities on the massive sulphide orebody; construction of the plant is to be completed during a period of one to two years; the mining period will be in accordance with the approved reserves and the granted mining license, and BPNM will implement exploration and feasibility studies for disseminated sulphide mineralization and other mineralization prospects within the 150 km^2 licensed area, together with mining of the massive sulphide ore.

- The Project will also be entitled to certain incentives whose scope is expected to be clarified in a pending amendment to the Investment Certificate as follows:

- corporate income tax rate of 15% until 29 January 2013 (i.e. 20 years from the IL issue date), after which the rate will be in accordance with the then existing regulations;
- an exemption from corporate income tax for four years after BPNM begins to make profits;
- a 50% reduction in corporate income tax for the further period of four years beginning with the end of the 4 years' exemption period; and
- import tax exemption for all machinery and equipment imported for the project.

The royalty tax rate of 4% is provided for under the Investment Certificate of BPNM and applies for the first 20 years from the date of the first IL, 29 January 1993 (i.e., it is available until 29 January 2013 only). After 29 January 2013, the royalty rate is 10% for nickel and associated minerals (i.e., copper and cobalt), in accordance with the Resolution 928/2010/UBTVQH12 dated 19 April 2010 of the Standing Committee of the National Assembly.

4.3.1 Mining Licence Amendment Process

BPNM's current mining licence, 17 December 2007, describes an annual ore throughput rate of 200,000 tonnes. BPNM is currently in the process of amending its Mining Licence to reflect the increased throughput rate targeted by its current mine plan (up to 360,000 tonnes of ore).

The Mining Licence amendment process can be summarized as follows:

- Submission of an amended Basic Design and EIA by BPNM to the relevant Vietnamese authorities for their review and comment;
- Revision of the amended Basic Design and EIA by BPNM to reflect any comments received during the review process. Upon satisfactory completion of the relevant Vietnamese authorities' reviews, the amendment to the Mining Licence will be recommended by the General Department of Geology and Minerals ("GDGM") to the Ministry of Natural Resources and Environment ("MONRE"); and
- Following completion of its review, MONRE will issue the amended Mining Licence, which will then come into force.

4.4 Environmental Liabilities

There are no environmental liabilities known at present (Section 20).

As required by law, and committed under the approved ESIA, BPNM will provide an environmental bond of US$ 240,000. This bond is refundable after BPNM completes rehabilitation as outlined committed in the ESIA upon closure of the mine.

4.5 Other Operating Permits

Following the grant of the Mining License, BPNM has submitted and will submit various permit applications to the Son La Provincial Government for approval. Major approvals for the Project are listed in Table 7.

Table 7 Details on major approvals for the Project.

Permit/ approval	Agency	Status	Comments
Mining Licence	Ministry of Natural Resources and Environment (MONRE)	Mining licence No. 2089/QD-BTNMT issued on 17 December 2007 permits BPNM to mine the 7ha massive nickel sulfide deposit at Ban Phuc, Muong Khoa, Bac Yen, Son La Province and is valid for 11 years until 16 December 2018.	BPNM is currently amending its mining licence to increase mining capacity from 200,000 tonnes per year to 360,000 tonnes per year, with process expected to be completed in April 2013.
Mine closure plan (MCP)	MONRE	No action required.	MCP needs to be submitted for approval 6 months prior to mine closure and approval obtained before the mine is closed. A preliminary conceptual MCP has been prepared and to be updated during the detailed engineering stage and periodically.
Investment Certificate No. 241022000033, dated 30 July 2007 (as amended)	Son La People's Committee	The investment certificate is valid until 29 January 2043, and has been amended five times on 30 August 2007, 30 July 2008, 10 October 2008 4 January 2011 and 22 November 2012.	BPNM is currently amending its investment certificate to reflect increase in total investment capital from US$87m to US$136m, which is expected to be completed in February 2013.
Reserve Evaluation and Assessment, Approval.	National Mineral Resource evaluation Council	Reserve Evaluation and Assessment Approval No. 563/QD-HDTL dated 14 June 2006 remains valid.	
Reserve and Resource Category Conversion	National Mineral Resource Evaluation Council	Reserve and Resource Category conversion Approval No. 01/CD-HDTL dated 27 September 2006 remains valid.	
Environmental & Social Impact	MONRE	Approval granted 25 December 2006 as No. 1942/QDBTNMT	BPNM is currently seeking approval of an amended EIA as part of the amendment to the Mining Licence.

Permit/ approval	Agency	Status	Comments
Assessment Report (EIA)			
Approval of Environmental Reclamation & Rehabilitation Project (ERRP)	MONRE	ERRP approved under Decision No. 1859/QD-BTNMT dated 6 October 2011.	BPNM is currently seeking approval of an amended ERRP as part of the amendment to the Mining Licence.
Rehabilitation Security Deposit	MONRE	The approved rehabilitation security deposit is VND4,992,049,000 (approx. US$243,515). BPNM has already paid VND2,995,229,000 (approx. US$146,109) for the years 2007-2012 certified by letter No. 13/QBVMT-XN dated 17 January 2012 by the Environmental Protection Fund of Son La Annual instalment of VND399 million (approx. US$19,200) is payable for the years 2013-2018 on or before 31 January of each year, which was also paid in January 2013.	
Final Resettlement Plan (RAP)	Son La Relocation and Resettlement Board	The Phase 1 (22 households and 2 organizations) and Phase 2 resettlements (89 households and 2 organizations) were completed in 2004 and 2007 respectively. The Phase 3 resettlement commenced in 2007 and is currently near completion. The total	No further relocation or compensation requirements are outstanding. The Phase 1 (22 households and 2 organizations) and Phase 2 resettlements (89 households and 2 organizations) were completed in 2004 and 2007 respectively. The Phase 3 resettlement commenced in 2007 and is currently near completion. The total compensation payout for the Phase 3 resettlement was approximately US $36,200, and was completed in November 2012.

Permit/ approval	Agency	Status	Comments
		compensation payout for the Phase 3 resettlement was approximately US $36,200, and was completed in November 2012.	
Land Lease Permits and Approvals	Son La Province	BPNM is leased with 1,072,972.5 m^2 of land for the mining area (7 ha), industrial auxiliary works, office and camp site, domestic water pipeline and auxiliary facilities, water station, pipeline and auxiliary facilities.	Application for land use rights over an area of 71,794.5 m^2 for storage of waste rock from mining activities.
Permit for Use of Industrial Explosives	MOIT	Permit for use of industrial explosives No. 02/GP-ATMT dated 19 March 2008 remains valid until 31 March 2013	Application for extension is in process.
Permit to Use Frequency and Wireless Equipment	Agency for Radio Frequency Management	Permit to Use Frequency and Wireless Equipment No. 143600/GP-GH dated 24 November 2011 remains valid until November 2013	
Registration of hazardous waste owner	Son La DONRE	Obtained on 10 December 2010	
Renovation of road located in the traffic lobby of the national road No.37.	Son La Department of Transportation	Decision 582/SGTVT-GPTC of Son La Department of Transportation dated 29 August 2008.	

Permit/ approval	Agency	Status	Comments
Renovation of 4 road-connection points to national highway No. 37.	Son La PC and Transport Department	Decision 2194/QD-UBND of Son La PC dated 10 October 2012 approving the master plan of road connection points to national highways (including national highway No. 37), and License No. 1253/GPTC-SGTVT of Transport Department approving renovation of road connection points to national highway No. 37.	
Fire Prevention and Fighting for Storage	Son La Police	Certificate of Fire Prevention and Fighting Appraisal No. 81/TD-PCCC dated 24 December 2007	
Fire Prevention and Fighting for Usage	Son La Police	Certificate of Fire Prevention and Fighting Appraisal No. 78/DK-PCCC dated 2 December 2011	
Fire Prevention and Fighting for Processing Plant	Son La Police	Application in process.	This fire prevention and fighting certification for the processing plant will be required prior to the plant commencing operation. The application has been lodged and is expected to be obtained in June 2013.
Permits for import and use of radioactive equipment	Vietnam Agency for Radiation and Nuclear Safety of the Ministry of Science and Technology	Application in process. Permit expected to be received in April 2013.	Permit is needed for an in-stream analyser in Processing Plant.
Registration of specialized machinery and equipment for import	Son La Department of Planning and Investment and Son La Department of Customs	Valid. List no. 01/HQDB dated 18 Oct 2007 approved the equipment for free tariff import	The registration will be updated for any additional equipment that is deemed necessary for import.

Permit/ approval	Agency	Status	Comments
Registration of date of mining	MONRE	Under BPNM's mining license, the company is required to register the commencement date for production with MONRE. BPNM is also required to notify Son La People's Committee, Bac Yen District People's Committee and Muong Khoa Ward People's Committee on the commencement date of production.	No deadline for registration. Notification will be carried out before commencement of mineral exploitation activity.
Construction Approvals (roads, plant, TSF, dams, power supply, admin., buildings, etc.)	Son La Province	No construction permit is required for the construction of the mine and its auxiliary facilities. Application for construction permit for waste dumping ground will be lodged upon receipt of land use right certificate and design document for the area.	BPNM obtained the appraisal of the basic design of the mine under Decision 178/SCN-KT of the Department of Industry of Son La province dated 28 March 2007. BPNM obtained Official Document 2521/UBND-KTN approving construction on 11 Oct 2007.
Mine design	GDGM, Son La Department of Natural Resources and Environment (DONRE) and Son La Department of Industry and Trade (DOIT)	BPNM submitted its mine design to the relevant authorities on 29 November 2010. GDGM then confirmed and approved the Mine Design by Official Letter No. 2379/DCKS-KS dated 01 December 2010.	Under Official Letter 705/DCKS-KSHDKS of GDGM dated 06 October 2011, GDGM concluded that the specifications of the Mine Design is not in line with the "exploitation investment project" (i.e. the F/S) and the EIA approved by the competent authorities. BPNM then submitted Report No. 68/2011/BPNM dated 7 November 2011 ("Report 68") and Letter No. 35/2012/BPNM dated 5 June 2012 to GDGM to explain the difference of specification of the Mine Design between the approved Mine Design and the technical specification in the approved F/S and ESIA. GDGM has had no further requirement on this matter.
Permits for Discharge of Waste Water, and Surface Water Use	MONRE/Son La Province	BPNM is currently in the process of applying for such permits.	

Permit/ approval	Agency	Status	Comments
Appointment of Mine Director	GDGM and Son La DONRE	BPNM informed the appointment of the Mine Director to relevant authorities by sending. Notice No. 66/2010/BPNM of the Company dated 05 October 2010.	
Export License	MOIT	Under Circular 41 of the MOIT dated 24 December 2012 on export of minerals, which shall take effect from 4 February 2013, BPNM is specifically included in Circular 41 to be entitled to export nickel concentrates of minimum 9.5% Ni grade without time limit.	
Approval for temporary suspension and extension of the project schedule	MONRE, MOIT and Son La Province	Approvals for temporary suspension: Official Letter 51/UBND-KTN of Son La PC dated 10 January 2009 and Official Letter 255/UBND-KTN dated 18 February 2009 of Son La Province; and Official Letter 4302/BCT-CNNg of dated 13 May 2009 the MOIT. Approvals for extension: Official Letter 149/DCKS-KSHDKS dated 14 February 2012 of GDGM; and Official Letter Official Letter 149/TB-UBD dated 06 September 2012 of Son La Province.	In its Official Letter 149/DCKS-KSHDKS, GDGM agreed to extend the completion of the mine construction until the end of 2012. In its Official Letter 149/TB-UBND, Son La Province agreed to further extend the production commencement date of the project to 30 June 2013 and BPNM is working towards completion of mine construction by end June 2013.
Registration of the Internal Labour Rules	Son La Department of Labour, Invalids and Social Affairs	Registered on 7 May 2010	
Registration of Health Safety Rules	Son La Department of Labour, Invalids and Social Affairs	Registered on 7 May 2010	

Permit/ approval	Agency	Status	Comments
Registration of the Collective Labour Agreement ("CLA")	Son La Department of Labour, Invalids and Social Affairs	BPNM is not required by law to have a CLA and has not been requested by any trade union to enter into a CLA.	

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The Ban Phuc deposit is located within rugged terrain of the mountainous areas in the north-west of Vietnam. The steep-sided Da River Valley traverses the region in a general south-easterly direction. On the northern side, steep mountainous country rises to about 1,200 m near Hong Ngai. On the south side of the Da River similar mountainous country rises to 1,520m.

Second order streams with catchments of 20 to 200 km^2 drain north and south into the Da River and commonly have small alluvial flats in their lower reaches. Third and fourth order streams in steep V-shaped valleys at close intervals drain into the second order streams and directly into the Da River. This closely spaced drainage system results in steep slopes of 30º to 45º, which are further exaggerated in areas of calcareous rocks, where upstanding massifs and karst topography occur, especially in the east and southwest.

Two hydroelectric dams are situated in the Da Valley. The Son La dam is located approximately 50 km upriver from the confluence of Chen Stream with the Da River. The other dam at Hoa Binh is located approximately 130 km downstream from the confluence of Chen Stream with the Da River. The dams regulate flows and the water level fluctuates from a low of about 104 m RL in the early to middle months of the wet season to a high of about 117 m RL in the early to middle months of the dry season.

The topography in the actual project area ranges between steeply sloping hills with elevations between 100 and 550 m RL and narrow valleys with few flat areas.

The Project is located in mountainous country (Figure 3) that contains areas of relatively degraded forest and localized bamboo and deciduous forest. Remnant forest areas are under increasing pressure from hunting activities, shifting cultivation and expanding settlement. Large areas of grassland, bamboo and other secondary vegetation are also present, particularly on the lower slopes and valleys.

Three forest communities have been identified within the project area:

- Re-growth secondary forest;
- Shrub interpolating timber; and
- Bamboo forest, the dominant ecosystem.

Figure 3. Ban Phuc Nickel Project area overview – looking to northwest explosives magazine in centre left and upper ROM pad centre right.

5.2 Access to Property

The Ban Phuc Nickel Project is situated approximately 160 km west of Hanoi in the Son La province, in the northwest of the Socialist Republic of Vietnam (Figure 1).

The Ban Phuc site is easily accessible. The provincial highway to Son La is 35 km by paved road from Ban Phuc. From Son La, access to Hanoi and the port at Hai Phong is by national highway. Alternate light vehicle access via Son Tay, Thanh Son, Phu Yen, Bac Yen and Ta Khoa provides a shorter travelling time from Hanoi on fair to good paved roads.

There is a local airport at Hat Lot, near Son La, which is closed for an upgrade and runway extension to make it jet capable. However, the local airport can be used for helicopter flights for medical evacuations and other emergencies.

5.3 Climate

The area essentially has two seasons: a dry season (winter) and wet (summer) season. Winter is cool and lasts from October to March with persistent drizzling rain occurring during February and March. Hot monsoonal summers occur between April and September with occasional typhoon events.

The meteorological data summary is based on data from Co Noi weather station (January 1964 to December 2008) located 20 km southwest of Ban Phuc and the Project weather station which was established in July 2004.This data is presented in Table 8.

Table 8. Ban Phuc Area Climate Data

Characteristic	Unit	Data
Dry Season (October - March)		
Lowest recorded temperature	°C	1
Mean temperature	°C	8
Mean relative humidity	%	79
Lowest relative humidity	%	76
Maximum recorded rainfall (Nov 1983)	mm/d	99
Prevailing wind direction		NE
Average wind velocity	m/s	1
Maximum wind velocity	m/s	20
Wet Season (April - September)		
Highest recorded temperature	°C	40
Mean temperature	°C	29
Mean relative humidity	%	83
Lowest relative humidity	%	79
Maximum recorded rainfall (September 2008)	mm/d	314
Prevailing wind direction		SE
Average wind velocity	m/s	1
Maximum wind velocity	m/s	30
Annual		
Evaporation (mean annual total)	mm	1,089
Rainfall (mean annual total)	mm	1,289

5.4 Infrastructure

5.4.1 Sources of power

A 35 kilovolt power transmission line runs from the Son La sub-station, some 40 km from the Project, to within 1 km of the Project. The Son La Provincial Government Power Department (PC Son La) has submitted a design for a 1 km spur line to the proposed 35kV/6.3kV substation. Power will be reticulated to the process plant, tails line pump stations and the underground mining operations through low voltage motor control centres.

5.4.2 Water

Process water will be recycled from the tailings storage facility (TSF) with make-up and raw water drawn from the Chen Stream which feeds into the Da River.

The camp will draw water from the Da River to supply a reverse osmosis plant for domestic, non-potable water use.

Drinking water is provided in bottles or purpose made containers.

5.4.3 Mining facilities

The Project is connected to the national grid via a fibre-optic connection. This provides telephone, facsimile links and broadband internet access. Internally, the camp and process plant site are connected with wireless LAN coverage.

Two-way radio communications are established with fixed, in-vehicle or hand-held units accessible to all personnel.

The camp is located 3 km from the mine site on the east bank of the Chen Stream downstream of its confluence with Dam Creek, on a site already leased by BPNM. The site is 35 m to 100 m wide and approximately 300 m long. Due to the limited area available, two storey accommodation blocks have been constructed, with up to five rooms on each level. This is in line with building practices in the area.

6 History

6.1 Property Ownership

The Ban Phuc Nickel Project area has a relatively long exploration history. The methods include drilling, trenching, cross-cutting, underground drives and channels. There is no record of previous production at Ban Phuc.

6.2 Project Results – Previous Owners

Initial work in the Ta Khoa region by Vietnamese and Chinese geologists focused on areas of known copper mineralization: Van Sai in 1959-61; Na Lui 1959-60; Ban Bo 1959-60; Na Ka in 1960-62; and Ban Phuc 1959-63. Follow-up reconnaissance work in 1961-1964 delineated several new zones of nickel, with or without copper, in nine areas and copper without nickel in an additional five areas.

To 2003 there are 154 drill holes totalling 18,741 m drilled into all prospects. There are also 169 adits, cross-cuts, drives and channels, totalling 5,107 m. Most nickel mineralization, with or without copper, is both spatially and temporally associated with ultramafic intrusions including:

- disseminated low-grade Ni or Ni-Cu mineralization (DISS) in basin shaped cumulate layers (locally multiple), often near the base and walls of ultramafic intrusions, e.g. Ban Phuc, Ban Khoa; and
- veins of high-grade massive Ni-Cu sulphide (MSV) in metasedimentary wall rocks adjacent to ultramafic intrusions, with locally developed low-grade disseminated Cu-Ni mineralization marginal to the MSV, e.g. Ban Phuc, Ban Trang, and Ban Mong.

7 Geological Setting and Mineralization

7.1 Regional Geology

The concession area lies in the Song Da rift, a major crustal suture zone, which is part of a broader northwest trending corridor of deep continental rifting known as the Red River Fault Zone (Figure 4). This feature acted as both a rift and collision zone between the South China Continental Plate to the north-east and the Indochina Plate to the south-west and is recognized as a rare case of where faulting has extended the entire thickness of the continental crust to the base of the lithosphere.

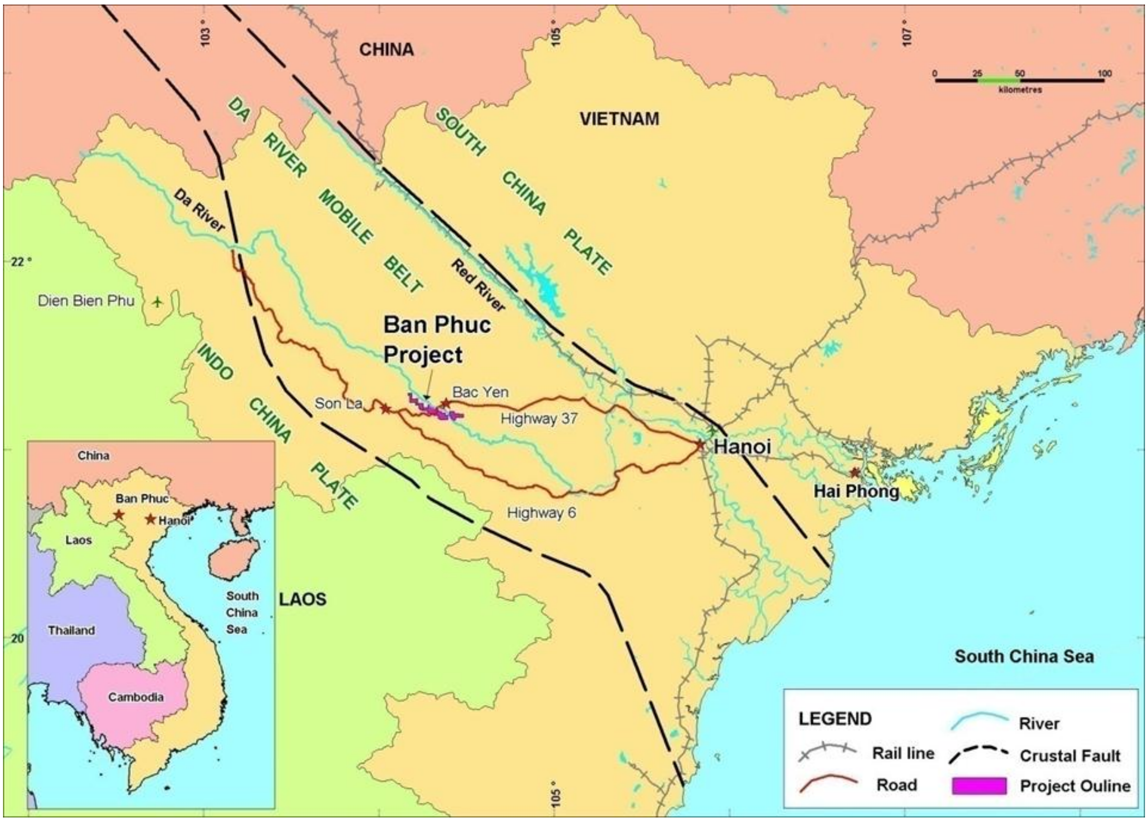


Figure 4. Regional Geological Setting

Within this faulted corridor, and in the Ta Khoa region, an anticline of Devonian limestones and terrigenous sediments is overlain by an unusual Permian-Triassic flood-basalt suite. The Devonian sequence is intruded by numerous ultramafic intrusions of compositions ranging from high-Mg gabbro, pyroxenite, and peridotite to dunite. The intrusions are interpreted as sub-volcanic dykes and sills representing feeders for the overlying volcanic suite.

Many of the intrusive have associated sulphides mineralization. Ban Phuc is emplaced close to the axial zone of the Ta Khoa Shoa anticline and is the only intrusive to date to have a quantified Ni-Cu sulphide resource.

The geotectonic setting is strongly analogous to that displayed by some major Ni-Cu deposits such as Norilsk and Jinchuan. Both of these are located on major breaks between lithospheric plates, associated with deep, mantle-tapping structure which allows the rapid ascent of mantle melts favoured for Ni-Cu sulphide segregation. At Norilsk in particular, the orebodies are interpreted to be associated with sub-volcanic intrusive bodies which represent feeders for extrusive flood basalts higher up in the sequence.

7.2 Prospect and Local Geology

The Ban Phuc ultramafic intrusion is one of the larger of such bodies in the district outcropping over an area of roughly a quarter of a square kilometre. The Ban Phuc ultramafic is exposed in a window comprising a basement metamorphic complex of Devonian age metasedimentary and metavolcanic rocks. The basement includes a variety of rock types including: quartz-feldspar schist, phyllite, metaquartzite, sericite schist, muscovite-biotite schist, calcareous schist and intercalated crystalline marble, phyllite, metaquartzite and sericite schist overlain by a thin unit of limestone (Figure 5).

Besides the Ban Phuc ultramafic body, Devonian age rocks exposed in the Ta Khoa window are intruded by numerous small bodies (about 70 have been mapped at 1:100,000 scale) of ultramafic to gabbroic (wehrlite) composition plus granite and granite pegmatite. Twenty-eight are of ultramafic composition. The intrusions are lensoid and up to about 3.5 km in length; generally they exhibit parallel bedding in the host metasediments. Both metasediments and intrusive have undergone tilting and folding, possibly during a Triassic orogeny.

The ultramafic-mafic intrusives are considered to be Triassic in age, although some are postulated to be lower to mid-Palaeozoic. At least some of the Triassic volcanics are understood to be extrusive equivalents of the ultramafic-mafic intrusive.

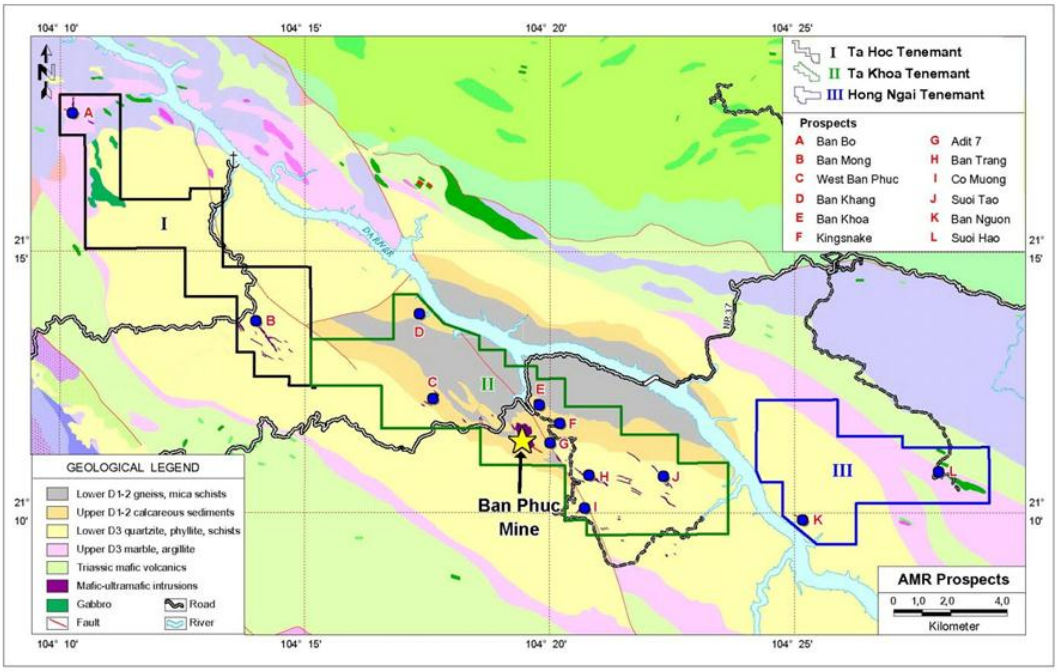


Figure 5. Ban Phuc Local Geological Map

8 Deposit Types

8.1 Mineralization Styles

The Ban Phuc intrusion is one of the larger ultramafic bodies in the region with dimensions of 940 m by 220 - 420 m, an outcrop area encompassing 0.25 km^2 and, a preserved depth of up to 470 m below surface. The intrusion is elongate with a north-westerly trend corresponding to the strike of the Devonian meta-sedimentary host rocks.

It has intruded along the trend of a discontinuous unit of crystalline limestone. At its wider north-western edge, only the flat lying base is preserved. The intrusion narrows and deepens to the southeast where it has an oval cross-section dipping steeply northeast and roughly concordant with the enveloping metasediments. A locally discordant contact with metasediments confirms the body is intrusive and not extrusive as in nickel deposits of the komatiite type.

Concave layering is defined by low-grade nickel-enriched sulphide layers which are conformable with the base and walls of the intrusion. In the wider basal zone preserved at the north-western end of the intrusion these are flat lying with only minor convexity, but in the south-eastern section the layers are tightly oppressed and strongly concave, extending up the footwall and hangingwall of the intrusion.

There is evidence in bedding attitudes along strike to the southeast of the intrusion for a synformal structure, suggesting that the strongly concave layering may arise from folding, with the ultramafic originally having formed a thinner sill roughly conformable with the sediments and now occupying the fold axis. The presence of cumulate sulphides along the base and walls of the intrusion indicate that it is upright, though folding, if it has occurred, is isoclinal and the hangingwall is overturned.

The MSV mineralization occurs in a major shear controlled vein structure in hornfels host rock along the southern margin of the Ban Phuc intrusion. The vein is approximately 730 m in length, with an inverted triangular form in plan, to at least 450 m below surface with an average width of 1.26 m. It has a north-westerly strike of 280° - 310° and a steep dip of 70° - 90° to the northeast, rarely to the southwest. This vein cuts lithological layering in metasediments at a low angle but appears conformable in section. Offshoots and bifurcations are minor and the vein is largely a singular structure.

A Vietnamese team explored Ban Phuc with Chinese advisors during 1959 - 1963. Surface and underground investigations entailed 12.5 km of drilling, 3.0 km of adits and crosscuts, 1.0 km of shafts, 6,900 m3 of trenching and over 8,000 assays for nickel and over 4,000 assays for copper. As well as can be determined, this work appears to have been carried out competently and to a high standard.

A number of types of mineralization are recognized in the host intrusive and surrounding metamorphic rocks (Figure 6):

- A structure containing massive nickel and copper sulphides within hornfels-schist and tremolite altered dikes in the southern contact aureole (MSV);
- Selected disseminated copper-nickel sulphide in hornfels- schist and tremolite altered dikes abutting the massive sulphides (disseminated sulphide envelope);
- Nickel sulphides in dunite near the base and walls of the intrusive (DISS 1 and DISS 2); and
- Nickel silicate as garnierite in serpentine vertically above nickel bearing dunites (DISS 3 and DISS 4).

The first three styles of mineralization are the ones of most economic interest.

8.2 Massive Sulphide Vein (MSV) Type Mineralization

MSV mineralization contains a number of minerals as follows (relative percentages are given in parentheses): pyrrhotite (70%), pentlandite (10%), chalcopyrite (5%), magnetite (4%), pyrite (3%), violarite (2.5%), siderite, ilmenite, sphalerite, galena (<1%), non-Opaques (5%). Pyrrhotite occurs in 1-3mm grains with fine exsolutions of pentlandite. Pentlandite occurs in granular masses with a grain size of 0.06-2mm, and as fine inclusions in pyrrhotite. Chalcopyrite forms irregular grain aggregates up to 3mm in size, and inclusions in pyrrhotite.

MSV occurrence at the Upper Portal underground Adit. Channel sample number and slots can be seen in the Figure 7.

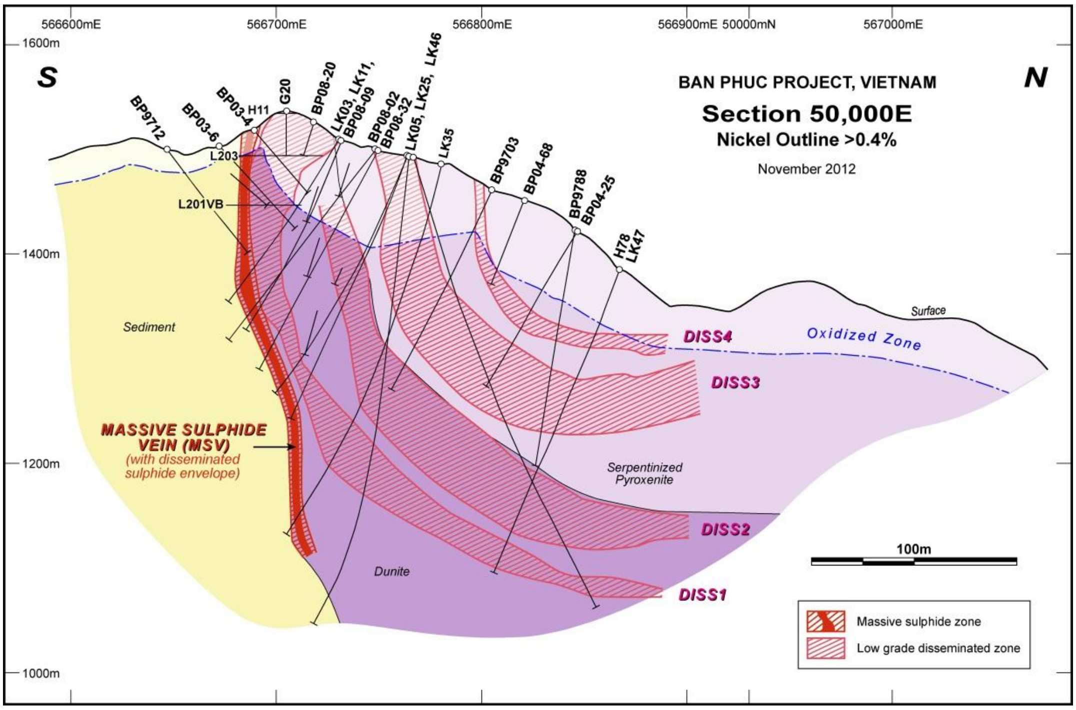


Figure 6 Ban Phuc mineralization styles

Figure 7. MSV occurrence at Upper Portal underground adit (450mRL) looking east width of vein is 0.8 m

8.3 Selected Disseminated Sulphide Type Mineralization surrounding MSV

Selected disseminated sulphide type mineralization occurs in the tremolite-altered dikes, schists and hornfels and forms a halo around the MSV type mineralization. It can vary from nothing to several meters in extent. The sulphides occur as veinlets, stringers, and disseminations. Percentages are as follows: pyrrhotite (25%); chalcopyrite (30%); violarite and pentlandite (15%); pyrite (10%); ilmenite, niccolite, galena, sphalerite, valerite (20%).

There are significant differences between the massive and disseminated types of mineralization, particularly in regard to the amount of pyrrhotite. There are more supergene varieties of nickel sulphides (e.g. violarite, millerite) in the disseminated mineralization.

8.4 Low Grade Disseminated Type Sulphides (DISS) in Dunite

Low grade disseminated sulphides (DISS) mineralization is present in the dunite within the Ban Phuc intrusive body. Grades in the range 0.5 to 1.0% are common, higher grades (i.e., 2 to 15 m at + 2%) appear to be localized. Minor chalcopyrite is also present in this type of mineralization. Nickel is present in both sulphide and silicate forms with the sulphide type being dominant.

8.5 Oxidized Type Mineralization

Oxidization of massive and disseminated types of mineralization has occurred near surface, typically to depths of 10 to 40 metres; nickel has been solubilized and leached while copper has been altered to malachite and other oxides (Figure 6 and Figure 8). The distribution of supergene nickel sulphides such as violarite and millerite is presently poorly understood.

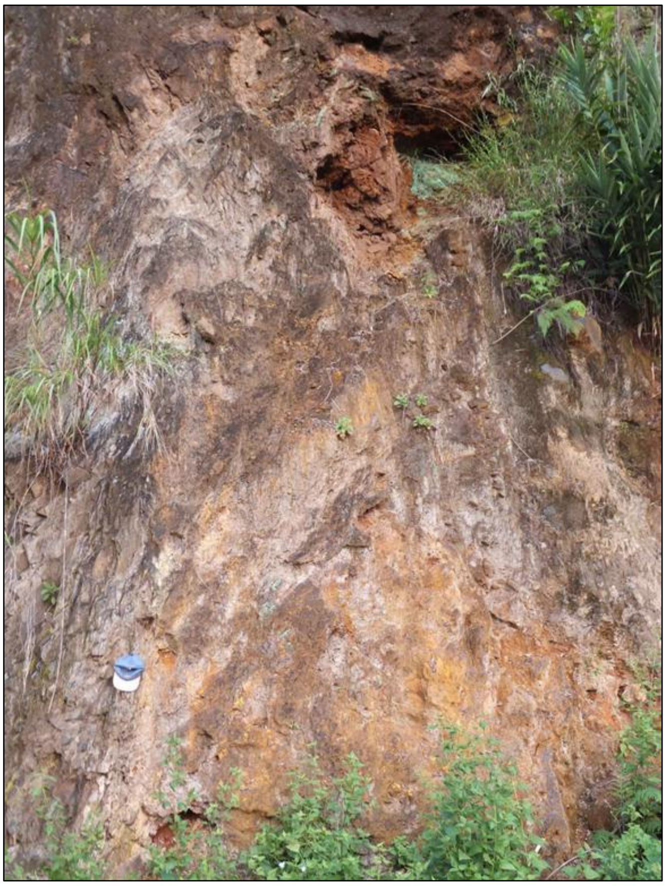


Figure 8 Outcrop of MSV at 450RL looking southeast.

**Note zone of malachite mineralization in upper centre. Width of zone at cap is approximately 2.5m*

9 Exploration

9.1 History

Initial work in the Ta Khoa region, by Vietnamese and Chinese geologists, focused on areas of known copper mineralization: Van Sai in 1959-61; Na Lui 1959-60; Ban Bo 1959-60; Na Ka in 1960-62; and Ban Phuc 1959-63. Follow-up reconnaissance work in 1961-1964 delineated several new zones of nickel and copper in 9 areas and copper in an additional 5 areas. There are 154 drill holes up to 2003, these total 18741 meters. 169 adits, cross-cuts, drives and channels exist, these total 5107 meters. A total of 76 holes were completed totalling 14,520 m in 2004.

9.2 2005 - 2010 Exploration Program

During 2005 to 2010, the exploration program was directed at exploring and gaining further information of the kink structure at the eastern zone of the MSV. Underground drilling conducted from adits on the 201, 202 and 301 RL, which was undertaken to upgrade resource confidence and enable mine design. In total, 75 holes underground were drilled for 2,837.9 m, additionally, 35 drill holes from surface were completed totalling 7,032.5 m.

10 Drilling

10.1 Summary of Drilling

All drilling on the Ban Phuc project area since 1996 has been done by InterGeo, a Vietnamese government company. The Falconbridge and spring 2003 programs employed a Longyear 38 and an equivalent Russian rig. Holes were fully cored (mostly HQ, some NQ wire-line size) with recoveries generally exceeding 80% in the oxidized zone and normally over 95% in fresh rock. All drill core was quartered with a diamond saw for sampling purposes.

10.2 Drilling and Survey Control

Available information, including field checks, indicate that the Ban Phuc area was surveyed between 1959 and 1962 with all drill collars being tied into a surveyed grid and levelled. Similar detailed surveying was carried out underground.

Before the 1996 drill program BPNM established a surveyed levelled grid marked with concrete survey points at 10 m intervals on lines spaced 50 m apart. A new baseline was positioned on the same bearing and as close as possible to the original Vietnamese baseline. Two reliable survey points and one drill hole from the old grid were located. During the BPNM survey a number of old drill pads, pits, trenches and adit portals were located and tied to the new grid. This, together with the matching of topographic points, indicates a very close correlation between the two grids.

Drill hole deviation in the vertical plane was recorded during the Vietnamese drilling.

This data shows considerable flattening in some holes at deeper levels but no deviation in the horizontal plane is recorded. Therefore, the accurate position of deeper holes must be regarded with suspicion. This does not affect the bulk of current resources but is an important consideration when attempting to predict the existence or continuity of the massive sulphide zone at depth for future estimates.

Note that the site grid established by BPNM is specific to the Project. It has been used for geology and underground mining purposes. The Vietnamese national grid VN2,000 is used for all other purposes.

10.3 2004 Phase 1 Drilling Program

The collar coordinates of phase 1 and other details are listed in Table 9.

A total of 28 holes were completed totalling 2,834 metres for Phase 1; 48 holes for 10,840m for Phase 2/3 and 6 holes for 1,205 metres for the remaining Phase 3 holes (Table 10 and Table 11).

Table 9. 2004 Phase 1 Drilling Program

HoleID	E	N	RL	Azimuth	Dip	Depth
BP04-01	50025.13	50072.75	501.2	180.33	50	151.3
BP04-01A	50025.07	50073.84	501.21	180.33	50	200.6
BP04-02	50150.29	50034.13	473.66	180.33	45	51
BP04-03	50124.71	50039.07	491.45	180.33	50	78.4
BP04-04	50099.54	49991.41	483.26	0.33	45	39.8
BP04-05	50176.04	49989.39	442.83	0.33	45	57.64
BP04-06	50175.72	49968.77	430.04	0.33	45	79.55
BP04-07	50074.82	50061.49	492.73	180.33	45	138.2
BP04-08	50200.42	49990.3	428.85	0.33	45	60
BP04-09	50125.32	49981.18	458.98	0.33	50	59
BP04-10	50025.17	49951.58	491.35	0.33	45	101
BP04-11	49975.07	50053.69	505.8	180.33	45	146.6
BP04-12	49875.57	50045.02	493.49	180.33	45	117.9
BP04-13	49974.88	49951.78	527.53	0.33	45	99.15
BP04-14	49925.1	49930.39	520.65	0.33	45	136.2
BP04-15	49874.92	49960.58	475.28	0.33	45	112.9
BP04-16	49924.89	50046.97	501.94	180.33	45	127.6
BP04-17	49825.32	49934.55	432.48	0.33	50	118.2
BP04-18	49800.1	50311.28	460.08	180.33	55	86.5
BP04-19	49824.94	50344.97	444.78	180.33	45	90
BP04-20	49825.04	50021	446.32	180.33	50	82.7
BP04-21	49900.44	50349.84	423.94	180.33	52	49.4
BP04-22	49848.88	50357.56	439.71	180.33	55	53
BP04-23	49600.16	50208.35	340.3	180.33	50	140
BP04-24	49500.02	50150.68	261.27	0.33	45	87.45
BP04-25	49399.18	50290.86	323.91	180.33	50	124.5
BP04-26	49449.76	50208.76	316.92	180.33	60	85
BP04-27	49550.01	50252.77	331.96	180.33	50	160.5
Total						2834.09

Table 10. 2004 Phase 2-3 Drilling Program

HoleID	E	N	RL	Azimuth	Dip	Depth
BP04-28	50050.12	50219.76	446.32	180.33	75	294
BP04-29	50123.69	49948.35	450.95	0.33	50	126.65
BP04-30	49850.01	50110.87	446.07	180.33	52	10.4
BP04-30A	49850	50110.5	446.98	180.33	52	211.8
BP04-31	50049.87	49899.03	465.45	0.33	60	211.5

HoleID	E	N	RL	Azimuth	Dip	Depth
BP04-32	50250.16	49968.17	409.02	0.33	57	112
BP04-33	50250.16	49968.17	409.02	0.33	67	129.2
BP04-34	50149.98	49930	425.51	0.33	56	232.8
BP04-35	49999.82	50289.96	422.8	180.33	80	225.8
BP04-36	50050.28	50219.69	446.33	0.33	85	393.7
BP04-37	50199.96	49926.08	395.26	0.33	52	153.75
BP04-38	49700.07	49995.16	354.33	0.33	45	42.8
BP04-39	49849.87	50144.57	429.32	180.33	51	250.9
BP04-40	50125	50087.92	473.11	180.33	61	162.5
BP04-41	49699.69	50095.8	405.07	180.33	53	139.9
BP04-42	49800.17	50345.94	431.86	180.33	65	313.6
BP04-43	50049.95	50218.26	446.32	180.33	52	320.4
BP04-44	50249.96	50074.57	424.32	180.33	51	173.9
BP04-45	49849.9	50188.63	422.44	180.33	57	305.1
BP04-46	49750.02	50121.37	406.61	180.33	55	179.5
BP04-47	49900.01	50168.94	441.5	180.33	55	318.4
BP04-48	49950	50324.05	393.64	180.33	53	469.7
BP04-49	49749.85	50121.93	407.14	180.33	70	247
BP04-50	49849.82	50188.69	422.47	180.33	64	108.9
BP04-50A	49849.82	50188.69	422.47	180.33	64	388.1
BP04-51	49950.11	50210.76	449.94	180.33	47	103.15
BP04-51A	49950.11	50210.76	449.94	180.33	49	308.3
BP04-52	49900.32	50169.51	441.51	180.33	72	389.4
BP04-53	49750.03	50200.82	395.36	180.33	60	305.7
BP04-54	49849.68	50189.2	422.3	180.33	83	266.8
BP04-55	49949.71	50253.75	425.76	180.33	57	191.2
BP04-55A	49950	50250	425.76	180.33	57	391
BP04-56	49700.16	50177.01	357.69	180.33	45	236.4
BP04-57	49850	50189	422.35	180.33	72	219.4
BP04-58	49925	50141.5	456.02	180.33	46	218.8
BP04-59	49700	50177.89	357.65	180.33	55	240.4
BP04-60	50050	50105	473.13	180.33	47	182.8
BP04-61	49850	50189	422.39	0.33	85	231.8
BP04-62	49875	50115	449.48	180.33	51	188
BP04-63	50000	50130	488.55	180.33	67	269
BP04-64	49750	50200	395.01	180.33	90	181.5
BP04-65	49825	50125	431.41	180.33	54	214.4
BP04-66	50075	50093	468.76	180.33	46	157.6
BP04-69	50025	50235	448.31	180.33	80	202.6
BP04-70	50050	50320	393.25	180.33	77	171
BP04-71	50025	50235	448.29	0.33	80	195

HoleID	E	N	RL	Azimuth	Dip	Depth
BP04-72	50175	50115	438.91	180.33	50	159.15
BP04-73	49825	50124	431.62	180	67	294.7
Total						10840.4

Table 11. 2004 Phase 3 Drilling Program

HoleID	E	N	RL	Azimuth	Dip	Depth
BP04-67	49775	50186	396.73	180.33	83	182.7
BP04-69	50025	50235	448.31	180.33	80	202.6
BP04-70	50050	50320	393.25	180.33	77	171
BP04-71	50025	50235	448.29	0.33	80	195
BP04-72	50175	50115	438.91	180.33	50	159.15
BP04-73	49825	50124	431.62	180	67	294.7
Total						1205.15

Core recovery for the 2004 program was good with an overall average of 97.4% (weighted by interval). 99.1% recovery was achieved in the MSV and 98.2% in the UB2 (chief host for DISS mineralization). 98.0% recovery resulted in UB2 intervals in excess of 0.4% Ni.

10.4 True Thickness of Mineralization

10.4.1 Massive Sulphide Domain

The results of drilling programs above showed that a significant nickel and copper mineralization with average 4.27% Ni and 1.55% Cu at average 4.5m thickness of Massive Sulphide Vein extended from surface to about 350m depth with individual samples ranging up to over 7% Ni. Table 12 shows significant intercepts as an example of Massive Sulphide Domain.

Table 12. Significant Intercepts from the 2004 Drilling Programs for Massive Sulphide Domain

HOLEID	FROM	TO	Thickness	NI (%)	CU (%)
BP03-7	130.30	133.35	3.05	6.00	1.62
BP03-9	129.09	133.18	4.09	3.65	1.77
BP03-11	140.17	144.35	4.18	5.24	1.33
BP03-12	106.35	111.80	5.45	3.48	2.14
BP04-09	36.80	41.25	4.45	3.97	1.03
BP04-12	86.65	89.00	2.35	1.38	1.79
BP04-17	71.75	74.30	2.55	5.65	1.07
BP04-20	30.50	37.85	7.35	6.55	1.65
BP04-20	50.55	67.91	17.36	5.38	2.31
BP04-29	79.90	93.88	13.98	4.18	1.48

HOLEID	FROM	TO	Thickness	NI (%)	CU (%)
BP04-40	133.24	135.27	2.03	2.59	0.80
BP04-47	264.20	268.20	4.00	4.04	2.25
BP04-49	228.20	236.10	7.90	5.07	1.74
BP04-51A	286.00	289.43	3.43	3.19	1.68
BP04-53	281.75	284.62	2.87	6.32	2.01
BP04-62	175.45	178.10	2.65	3.91	2.04
BP04-63	258.73	260.75	2.02	4.64	3.60
BP04-65	199.20	201.65	2.45	5.06	1.03
BP08-02	193.25	195.90	2.65	2.42	0.59
BP08-10	70.80	74.50	3.70	6.08	1.16
BP08-14	124.15	128.40	4.25	6.81	1.88
BP08-18	156.35	159.60	3.25	6.48	0.66
BP201-08	10.20	14.00	3.80	6.24	1.22
BP201-19	7.00	10.00	3.00	7.10	1.34
BP201-20	2.50	7.70	5.20	5.23	1.15
BP201-21	2.60	7.30	4.70	4.21	1.15
BP202-67	27.90	35.00	7.10	2.17	1.17
BP202-75	32.90	35.05	2.15	3.86	1.62
BP202-75	38.00	41.80	3.80	1.78	0.50
BP301-4	17.90	21.00	3.10	6.86	2.21
BP301-5	20.55	24.50	3.95	5.63	1.48
BP301-6	46.90	52.50	5.60	6.59	1.70
BP301-17B	42.58	44.86	2.28	4.63	1.15
BP301-18	48.30	54.70	6.40	5.77	1.67
BP301-18A	34.35	36.95	2.60	6.70	2.34
BP301-19	22.40	26.00	3.60	5.65	1.81
BP301-21	12.86	15.30	2.44	2.27	0.98
BP301-23	32.30	35.35	3.05	4.68	2.10
BP301-23	36.50	43.40	6.90	5.46	1.78
BP301-24	12.80	15.90	3.10	4.09	0.58
BP301-25	28.35	31.00	2.65	5.88	1.84
BP301-25	34.60	48.00	13.40	4.30	1.90
BP301-26	21.80	24.30	2.50	3.98	0.94
BP301-28	18.76	22.15	3.39	7.25	1.59
BP301-29	15.85	19.65	3.80	4.49	2.76
BP301-30	24.00	30.40	6.40	4.45	1.93
BP9602	86.70	94.00	7.30	4.68	1.95
BP9603	36.60	39.90	3.30	3.41	0.91
BP9606	113.55	117.00	3.45	3.76	1.61
BP9610	78.20	82.00	3.80	2.85	2.14
BP9702	111.00	116.55	5.55	3.03	1.68

HOLEID	FROM	TO	Thickness	NI (%)	CU (%)
BP9702	116.75	119.00	2.25	2.01	0.73
BP9705	33.10	40.00	6.90	5.91	1.85
BP9712	122.10	126.65	4.55	1.53	0.90
L201IB	6.30	8.30	2.00	1.44	1.51
L201IIA	5.20	8.20	3.00	4.20	1.56
L301IIB	1.20	5.80	4.60	3.67	2.60
L301IIBW	1.20	3.20	2.00	1.86	2.03
L301IIIA	4.50	6.50	2.00	1.59	0.84
L301IVB	30.00	36.70	6.70	3.54	1.00
L301IVBE	29.00	36.70	7.70	2.80	1.29
L301VB	9.30	15.10	5.80	2.31	1.65
L301VBE	9.20	14.80	5.60	4.75	2.16
L302IVB	13.75	16.00	2.25	3.60	1.42
L401IIA	0.60	5.80	5.20	2.79	0.83
L401IIAE	0.60	4.80	4.20	6.01	1.58
L401IVA	1.00	13.40	12.40	3.61	1.48
LK02	109.00	113.75	4.75	3.57	1.76
LK03	167.94	170.44	2.50	3.98	0.96
LK11	188.60	192.95	4.35	2.80	1.08
LK16	287.90	291.05	3.15	2.23	1.05
LK27	130.05	132.45	2.40	4.20	2.60

10.4.2 Disseminated Sulphide Domain

The drilling results also showed that a significant nickel and copper mineralization with average 1.31% Ni and 0.24% Cu at average 6.2m thickness of Disseminated Sulphide extended from surface to about 350 m depth with individual samples ranging up to over 2.1% Ni. Table 13 shows significant intercepts as an example of Disseminated Sulphide Domain.

Table 13. Significant Intercepts from the 2004 Drilling Programs for Disseminated Sulphide Domain

HOLEID	FROM	TO	LENGTH	NI	CU
BP03-1	61.00	64.30	3.30	1.17	0.14
BP03-3	37.00	39.80	2.80	1.29	0.13
BP03-4	24.50	27.50	3.00	1.25	0.32
BP03-6	50.50	56.50	6.00	1.23	0.25
BP03-7	4.00	7.00	3.00	1.12	0.13
BP03-8	7.80	9.90	2.10	1.61	1.21
BP03-8	48.50	51.50	3.00	1.25	0.19
BP03-8	53.00	57.70	4.70	1.15	0.20

HOLEID	FROM	TO	LENGTH	NI	CU
BP04-10	79.20	82.60	3.40	1.51	0.31
BP04-11	9.00	13.00	4.00	1.21	0.15
BP04-11	32.00	34.00	2.00	1.13	0.13
BP04-13	54.00	58.00	4.00	1.24	0.24
BP04-13	60.00	67.00	7.00	1.25	0.25
BP04-16	4.90	8.00	3.10	2.19	0.46
BP04-19	0.00	8.60	8.60	1.73	0.25
BP04-19	8.80	22.10	13.30	1.34	0.21
BP04-22	0.00	11.00	11.00	1.86	0.13
BP04-36	149.00	153.00	4.00	1.09	0.21
BP04-36	166.00	169.00	3.00	1.33	0.21
BP04-42	263.00	265.00	2.00	1.08	0.17
BP04-54	215.00	219.00	4.00	1.28	0.23
BP04-55	59.30	62.00	2.70	1.46	0.29
BP04-55A	257.80	262.00	4.20	1.18	0.28
BP04-60	101.00	105.00	4.00	1.26	0.19
BP04-64	118.00	120.00	2.00	1.24	0.14
BP04-64	122.00	129.00	7.00	1.35	0.22
BP04-67	69.00	72.00	3.00	1.35	0.19
BP04-67	113.00	115.00	2.00	1.20	0.04
BP04-67	130.00	132.00	2.00	1.84	0.26
BP04-67	135.00	143.00	8.00	1.36	0.19
BP04-67	146.00	152.00	6.00	1.20	0.22
BP04-68	91.00	97.00	6.00	1.36	0.24
BP04-68	100.00	102.00	2.00	1.16	0.24
BP04-68	105.00	121.00	16.00	1.24	0.25
BP04-68	124.00	142.00	18.00	1.27	0.24
BP07-01	71.00	74.00	3.00	1.18	0.27
BP07-02	74.00	78.00	4.00	1.17	0.51
BP07-03	91.00	93.00	2.00	1.15	0.06
BP07-06	24.00	38.00	14.00	1.14	0.30
BP07-06	50.00	52.00	2.00	1.04	0.23
BP08-05	166.00	168.00	2.00	1.45	0.20
BP08-09	82.00	98.00	16.00	1.61	0.27
BP08-09	101.00	104.00	3.00	1.15	0.08
BP08-12	55.00	59.00	4.00	1.20	0.26
BP08-12	69.00	74.00	5.00	1.14	0.39
BP08-12	79.00	82.00	3.00	1.58	0.46
BP08-15	60.00	68.00	8.00	1.28	0.16
BP08-16	15.50	18.00	2.50	1.30	0.06
BP08-16	22.60	27.30	4.70	1.31	0.37

HOLEID	FROM	TO	LENGTH	NI	CU
BP08-16	32.00	43.60	11.60	1.25	0.25
BP08-18	56.00	62.00	6.00	1.28	0.14
BP08-18	68.00	79.00	11.00	1.30	0.21
BP08-18	85.00	88.00	3.00	1.18	0.17
BP08-18	99.75	102.00	2.25	1.27	0.26
BP08-18	104.00	107.00	3.00	1.19	0.26
BP08-20	84.00	89.00	5.00	1.19	0.52
BP08-20	99.00	115.00	16.00	1.25	0.22
BP08-20	127.00	131.00	4.00	1.17	0.18
BP202-02	11.00	16.00	5.00	1.22	0.20
BP202-02	17.00	20.00	3.00	1.08	0.32
BP202-09	16.00	18.00	2.00	1.29	0.52
BP202-17	25.40	28.80	3.40	1.46	0.35
BP9606	10.00	20.00	10.00	1.55	0.30
BP9607	34.00	54.00	20.00	1.38	0.19
BP9607	57.00	59.00	2.00	1.01	0.11
BP9607	63.00	66.65	3.65	1.35	0.12
BP9607	85.00	99.00	14.00	1.79	0.22
BP9706	130.00	136.00	6.00	1.54	0.22
BP9706	140.00	162.00	22.00	1.67	0.29
BP9714	165.00	167.20	2.20	1.43	0.19
L201IVA	16.20	20.20	4.00	1.02	0.28
L201IVA	25.20	29.20	4.00	1.38	
L201IVA	42.20	44.20	2.00	1.56	
L202IVA	14.50	16.50	2.00	1.12	
L202IVA	17.50	26.30	8.80	1.20	0.20
L202IVA	52.30	55.30	3.00	2.07	
LK05	0.00	5.70	5.70	1.11	0.07
LK25	299.00	301.00	2.00	1.29	
LK25	304.00	307.00	3.00	1.14	0.16
LK26	177.00	180.10	3.10	1.16	
LK26	195.17	199.24	4.07	1.92	
LK30	176.05	187.35	11.30	1.56	0.10
LK37	294.44	296.84	2.40	1.22	
LK38	204.45	211.45	7.00	1.43	
LK38	216.05	218.35	2.30	1.02	
LK41	124.10	131.10	7.00	1.12	
LK43	81.70	83.85	2.15	1.04	
LK43	114.35	116.40	2.05	1.12	
LK45	225.40	227.40	2.00	1.13	
LK46	162.92	188.40	25.48	1.74	

HOLEID	FROM	TO	LENGTH	NI	CU
LK46	193.90	217.13	23.23	1.41	
LK47	71.00	78.55	7.55	1.19	
LK50	117.05	131.40	14.35	1.40	
LK50	141.60	177.60	36.00	1.48	
LK51	115.55	117.85	2.30	1.06	
LK52	168.75	173.41	4.66	1.27	
LK52	175.05	177.95	2.90	1.09	

10.4.3 2005-2010 Drilling Program

Drilling was conducted from adits on the 201, 202 and 301 RL. In total, 75 holes underground were drilled for 2,837.9 m, additionally, 35 drill holes from surface were completed totalling 7,032.5 m.

For surface drill holes, the average core recovery of the 35 drill holes completed was 87.3%. Seven drill holes from the 2007 program were surveyed by using the Chinese dip and strike measurement instrument, whilst the remaining 28 holes were surveyed using a GyroSmart instrument.

Two drill holes in 2008 were collared but abandoned due to Typhoon Hapugit which destroyed them.

Underground drill average core recovery was 74.2%. Only collar surveys were performed as the drill holes were short.

The collar coordinates and other details are listed in Table 10.

Table 14 2005 – 2010 Drilling Program

HOLEID	E	N	RL	Azimuth	Dip	Depth
BP07-01	79948.52	49949.96	1544.17	0	-54	110.00
BP07-02	79925.74	50024.90	1482.43	0	-45	187.00
BP07-03	80081.03	50062.57	1477.96	180	-45	152.30
BP07-04	79926.87	49974.22	1521.49	0	-45	128.00
BP07-05	79952.72	49700.12	1340.06	0	-65.3	200.40
BP07-06	80022.86	50024.65	1525.66	120	-57	171.00
BP07-07	80044.61	50037.99	1510.67	120	-50	119.60
BP08-01	80098.77	49925.00	1468.44	180	-46.51	184.50
BP08-02	80097.04	50000.06	1498.67	180	-51.74	229.30
BP08-03	80054.30	50022.41	1510.51	120	-54.07	192.00
BP08-04	80065.55	49949.34	1497.33	120	-58.96	285.30
BP08-05	80100.77	50024.77	1487.34	180	-49.96	263.80
BP08-06	80078.56	49974.77	1497.40	180	-45.24	173.00
BP08-07	80076.17	49984.89	1497.74	120	-52.63	265.80
BP08-08	80101.31	49975.08	1487.94	180	-45.09	198.50

HOLEID	E	N	RL	Azimuth	Dip	Depth
BP08-09	80064.17	50004.69	1508.29	120	-53.1	206.90
BP08-10	79926.25	49900.00	1497.07	0	-45.08	173.00
BP08-11	79979.11	49900.05	1498.81	0	-43.23	74.40
BP08-12	80031.23	50067.31	1513.87	209	-45.14	140.40
BP08-13	79995.97	49890.11	1499.81	0	-44.53	82.00
BP08-14	80044.85	49925.00	1503.67	180	-59.33	129.40
BP08-15	80031.56	50067.53	1513.83	209	-56.92	154.80
BP08-16	79992.53	50024.92	1513.70	0	-44.73	64.70
BP08-17	80053.63	49970.30	1505.32	120	-57.35	238.00
BP08-18	80031.67	50067.59	1513.49	209	-66.89	180.80
BP08-19	80065.01	49951.66	1497.45	180	-51.57	165.70
BP08-20	80035.70	49995.42	1524.19	120	-57.84	212.20
BP08-21	80096.02	49949.31	1479.22	120	-53.4	268.88
BP08-22	80079.02	49925.84	1479.61	120	-57.2	256.00
BP08-32	80099.28	49998.99	1497.27	180	-59.95	237.40
BP08-38	80108.38	50117.29	1453.98	180	-65	38.80
BP08-39	79927.41	50150.00	1425.18	0	-51	35.20
BP09-01	80112.89	49725.03	1406.33	180	-75	281.10
BP09-02	80150.11	49600.00	1326.95	0	-90	399.10
BP10-01	80096.59	49649.77	1380.96	180	-80	264.70
BP10-02	80168.77	49697.38	1360.71	180	-68	270.10
BP10-03	80196.48	49649.88	1345.11	180	-62	372.40
BP201-06	79996.30	49862.56	1445.57	180	45	14.50
BP201-07	79983.04	49862.50	1443.78	180	0	17.10
BP201-08	79995.71	49862.46	1442.70	180	-45	24.00
BP201-18	79969.41	49912.40	1446.86	0	60	21.00
BP201-19	79969.80	49912.40	1446.54	0	48	20.30
BP201-20	79970.16	49912.52	1445.70	0	30	22.10
BP201-21	79970.39	49912.46	1444.75	0	0	16.50
BP201-22A	79970.69	49912.48	1443.82	180	-25	25.00
BP201-22B	79970.68	49912.48	1443.82	0	-25	38.20
BP201-28	79968.14	49925.30	1444.74	0	0	16.20
BP201-35	79966.53	49937.43	1444.82	0	0	18.00
BP201-47	79961.47	49962.51	1445.04	0	0	15.00
BP201-54	79960.21	49975.02	1445.32	0	0	16.50
BP201-63	79959.98	49987.49	1445.46	0	0	15.00
BP202-02	79991.80	50012.60	1446.77	180	40	48.00
BP202-03	79991.64	50012.59	1446.55	180	29	12.00
BP202-09	79998.89	50025.00	1446.97	180	38	48.30
BP202-10	79998.80	50025.05	1446.51	180	21	40.80
BP202-11	79998.74	50025.03	1445.56	180	-5	45.00
BP202-17	79995.77	50037.37	1447.87	180	46	39.00

HOLEID	E	N	RL	Azimuth	Dip	Depth
BP202-18	79995.31	50037.50	1446.83	180	29	45.00
BP202-20	79995.70	50037.44	1444.69	180	-23	46.00
BP202-21	79996.58	50037.42	1444.32	180	-46	45.00
BP202-25	80001.06	50050.10	1446.93	180	38	41.50
BP202-26	80000.88	50050.05	1446.05	180	20	44.70
BP202-28	80000.65	50050.02	1444.67	180	-23	49.80
BP202-42	80014.19	50074.98	1445.54	180	19	46.00
BP202-44	80013.94	50074.98	1444.60	180	-10	32.30
BP202-45	80013.94	50074.97	1443.78	180	-30	34.40
BP202-48	80014.50	50075.01	1446.52	180	40	43.00
BP202-49	80013.27	50085.26	1445.16	180	24	45.00
BP202-51	80013.11	50085.22	1443.73	180	-24	50.00
BP202-56	80023.75	50100.05	1445.81	180	51	39.00
BP202-57	80023.11	50099.97	1445.30	180	29	40.00
BP202-59	80022.00	50100.07	1442.80	180	-21	45.00
BP202-65	80025.63	50113.64	1445.79	180	33	25.00
BP202-67	80025.64	50113.63	1444.09	180	-31	46.00
BP202-72	80034.91	50136.66	1444.96	180	-24	32.00
BP202-74	80035.00	50136.60	1443.33	180	-31	50.40
BP202-75	80035.47	50136.55	1442.94	180	-45	46.00
BP202-76	80034.55	50136.67	1444.02	180	-5	21.40
BP202-77	80034.59	50136.70	1443.87	180	-10	40.00
BP301-1	80036.82	49761.48	1392.45	180	-45	24.70
BP301-2	80036.48	49761.63	1394.67	180	35	14.80
BP301-4	80036.82	49761.48	1392.45	180	-60	37.30
BP301-5	80010.77	49777.89	1393.91	0	-45	33.90
BP301-6	80011.06	49777.78	1393.93	0	-68	58.20
BP301-7	80020.68	49787.42	1393.68	0	0	29.70
BP301-9	80019.34	49787.43	1392.40	0	-70	64.90
BP301-11	80005.11	49801.86	1395.52	0	45	34.80
BP301-12	80005.17	49801.90	1392.68	0	-42	38.00
BP301-13	80004.83	49801.91	1392.66	0	-53	45.00
BP301-14	80004.55	49801.88	1395.73	0	63	33.40
BP301-15	80000.01	49812.64	1396.29	0	57	29.00
BP301-15B	80002.38	49812.46	1392.76	180	-57	36.00
BP301-16	80001.87	49812.54	1396.60	0	81	27.50
BP301-17	80000.28	49812.52	1392.80	0	-55	11.30
BP301-17B	80000.30	49812.56	1392.82	0	-65	49.80
BP301-17C	80002.62	49812.48	1396.47	180	55	26.90
BP301-18	80001.62	49812.52	1392.82	0	-80	70.30
BP301-18A	80017.77	49824.95	1393.99	180	0	56.90
BP301-19	80018.30	49824.84	1395.57	180	38	52.50

HOLEID	E	N	RL	Azimuth	Dip	Depth
BP301-20	80019.00	49824.97	1392.82	180	-38	71.00
BP301-21	80018.21	49824.92	1392.81	180	-56	58.00
BP301-22	80017.91	49837.52	1394.34	180	0	55.00
BP301-23	80018.23	49837.53	1395.12	180	26	52.00
BP301-24	80017.53	49837.50	1393.25	180	-25	45.00
BP301-25	80018.39	49837.59	1396.02	180	42	57.00
BP301-26	80018.41	49837.55	1392.95	180	-47	50.00
BP301-27	79997.77	49849.38	1396.01	180	50	36.50
BP301-28	79997.25	49849.37	1395.36	180	32	39.00
BP301-29	79997.32	49849.36	1393.28	180	-45	31.00
BP301-30	80011.59	49862.50	1394.33	180	0	57.80
BP301-32	80011.41	49862.52	1393.04	180	-24	53.40
BP301-34	80011.41	49862.51	1393.46	180	-44	37.30

11 Sample Preparation, Analyses and Security

At the commencement of drilling in 2004 and 2010 a new QA-QC program was implemented to ensure that the accuracy and repeatability of sample results being reported by Genalysis Laboratory Services (Genalysis) were of a standard to be used in feasibility style resource estimation.

The logging and sampling procedures were designed to achieve the dispatch of samples to the laboratory as quickly as possible after completion of the drill hole without compromising the quality of logging and sampling.

11.1.1 Drill hole logging and Sampling Sequence

1. A summary log is produced during and immediately after completion of the hole.

2. At regular intervals, summary drill logs and interpreted drill hole sections (development in process) are sent to the Hanoi office.

3. Geological logging is carried out. The sections of core to be sampled are finalised with input from the Project Geologist and Exploration Manager. A decision as to where Standard and Blank samples should be inserted were made at this time.

4. The logging/sampling intervals are established and core marked up. Note that the logging interval for the geotechnical log is on a drill run basis and the geology log is on an assayed core sample interval basis. Drill core is then photographed using a digital camera.

5. Handwritten drill logs are entered into the site computer using an ACCESS form. This is the primary database. Copies of the database and regular updates will be sent to the Ban Phuc Nickel Mines office in Hanoi, or any one that requests a copy. The site computer will have the most up to date database at all times. The supervising geologist will be responsible for the quality of the data entry.

6. Geotechnical logging is carried out before core cutting.

7. The core is cut in half (unless special treatment is requested in the case of, for example, metallurgical samples), and then one of the halves is quarter cored with the diamond saw or in the case of soft material with a knife or spatula. SG determinations must be made of every sampled interval. Samples after preparation at the project facility, including standards and blanks are bagged, labelled and dispatched to the assay laboratory.

8. Sample preparation consists of drying at 105 – 110°C for overnight (or 8 hours), followed by jaw crushing, roller crushing and pulverising. Detailed notes on sampling and storage of samples are available

11.2 Sampling

Sampling methods used at Ban Phuc include:

- surface trench sampling;
- underground adit channel sampling, and
- diamond drill coring.

11.2.1 Surface Trench Sampling

Exploration trenching was conducted during the period 1959 to 1960. A total of 78 trenches were completed, of which 46 were in the massive sulphide zone. The trenches were laid out at 25 m intervals along the strike of ore zone.

The trenches are mainly 1 m wide, 3.15 m to 4.4 m deep and 4.5 m to 85 m long. Samples were taken by channel sampling method mainly along the bottom of the trench, with the sample being 1 m long, 10 cm wide and 5 cm deep.

11.2.2 Adit Sampling

A total of over 3,000 m of exploration adits were excavated during the period 1959 to 1960. Four adit levels were used for massive sulphides exploration, the:

- 101-102-103 level;
- 201-202 level;
- 301-302 level; and
- 401-402 level.

The adits were developed along strike and in the foot wall of the ore body. Each adit has several cross-cuts at 50 m spacing. The adits range in length from 50.9 m to 198.7 m and the cross-cuts are from 4 m to 81.3 m in length cutting perpendicular to the massive sulphide vein and some reaching to contact between UB2 and UB1 in the ultramafic intrusion.

Samples were taken by channel sampling along the western wall of the crosscuts, each sample being 1 m long, 10 cm wide and 5 cm deep. The channels can still be seen on the walls of adits.

11.2.3 Diamond Drill Core Sampling

Diamond drill holes during the period 1959 to 1962 were 110 mm and 91 mm in diameter (approximating PQ and HQ sizes).

Drilling in the period 1996 to 2004 was mainly by Boart Longyear rigs drilling HQ (64 mm) and NQ (49 mm) core.

Drill cores were cut by electrical saws, first in half and then quarter core. Samples of quarter core were taken in lengths ranging from 0.2 m to 2 m long, but mainly 1m long. The average weights of the 1 m long HQ core samples were:

- 2.8 kg for massive sulphide ore;
- 1.5 kg for metasediments;
- 1.2 kg for tremolite dyke;
- 1.7 kg for UB1 rocks; and
- 1.8 kg for UB2 rocks.

Drill core was sampled where core contained greater than 2% sulphides. Normally sampling included the massive sulphide veins or mineralization zone and a further 15m either side of the mineralized zone.

Half drill core was selected for MSV metallurgy samples.

11.3 Sample Preparation, Analyses and Security

In 1989, a thorough review was made of sampling and sample preparation procedures used by the Vietnamese (Allen, 1989). As described, preparation by crushing and finally grinding of a 0.25mm fraction was carried out on site. As the splitting of samples was carried out progressively through the process there was a possibility that the final ground sample was not truly representative of mineralization.

Assay techniques employed in recent work use modern techniques in modern labs. In order to compare early Vietnamese sampling results with current procedures 28 metres of channel samples were collected from the walls of crosscuts iiA, iiiA and ivA on level 201 and crosscuts ivB in Adit 202 for AMR (described in Coote, 1989). This sampling variously cut massive nickel-copper mineralization; marginal disseminated nickel-copper mineralization in hornfels and disseminated cumulate sulphides in serpentinized dunite.

Channel width and depth dimensions were identical to those used in the 1959-63 sampling. The early sampling had been performed in a meticulous manner and was well marked making it possible to match the new sampling with the old. Assaying of the AMR collected samples for Ni, Cu and Co was performed in New Zealand using AAS (Table 15). Results from the two sets demonstrate comparable ranges to the 1959-1963 sampling as shown below:

Table 15. Assaying of the AMR collected samples

	AMR 1989	1959-1964
Nickel	0.05-6.30%	0.22-5.72%
Copper	0.10-2.50%	0.00-3.39%

This data indicates that for nickel values below 2% Ni, AMR values are lower than those of the initial sampling and for values above 2.5% Ni, AMR values are higher. Variations also occurred in copper but no systematic pattern was detected.

Overall, it has been demonstrated that the 1959 through 1963 assay data can be safely incorporated into a database for resource estimations. This conclusion has been supported by the similarity of values obtained from mineralization intersected in the 1996-97 drilling program to the 1959-1964 results from the same zones.

A representative random set of samples from quartered diamond drill core was collected in December 1996 and assayed by Chemex Labs in Vancouver (Leighton, 1997). The objective was independent verification of sampling results and analytical data being published by AMR. No significant discrepancies were found. In the opinion of Leighton (2003), AMR's sampling, sample preparation, sample security and procedures met or exceeded the then industry standards with the view taken that the Company had maintained on ongoing program of submitting duplicate samples to different laboratories as a method of cross checking analytical results. In the opinion, however, of Hellman & Schofield Pty Ltd (H&S), the QA-QC program by AMR was inadequate due to the absence of included standard reference materials or blanks. This was rectified for the 2004 drilling programs.

Since 1995 the following ISO accredited laboratories have been employed to assay stream sediment, soil, rock chip, channel and drill core samples:

- 1995-1997: BSE[1]/Analabs Ltd. (A joint venture between Australian, Hong Kong and the Vietnamese government);
- 1997-2001: Chemex Labs (North Vancouver, BC);
- 1997: Acme Analytical Laboratories Ltd. (Vancouver, BC);
- 2000-2002 Lakefield Research Limited (Ontario, Canada);
- 1993-1994, 2003-2004 Genalysis (Perth, Western Australia); and
- ALS-Chemex (Townsville, Queensland).

11.4 Sample Preparation and Security

The following procedures for QA-QC were implemented with the Ban Phuc Drilling commencing in February 2004.

11.4.1 Standards

One standard of known value, one coarse and one fine blank was included with the core samples per twenty five samples (i.e., each batch of 25 samples included 22 core samples, one coarse blank, one fine blank and one certified Ni, Cu and Co standard).

Four standards representing a low-Ni grade (0.86% Ni), a medium-grade (2.09% Ni) and two high-grade values (4.16% and 4.55% Ni) have been interspersed through the sampled sequences. The medium-grade sample (OREAS_14p) is a massive sulphide matrix standard from the West Musgrave block in Western Australia. The standard is certificated for Ni, Cu and Co and has an expected value of 2.09% Ni. The low-grade sample (G_MHB1) with a grade of 0.86% Ni is made from disseminated NiS material from the Maggie Hays Mine in Western Australia. There are two high-grade standards used during the program. One (G_M4) has a grade of 4.55% Ni and the second a (OREAS_M3) has a grade of 4.16% and both are sourced from the Miitel Mine. The submission of standards of known value monitors the accuracy of assays.

[1] The ISO status of this laboratory is unknown

11.4.2 Blanks

The coarse blank consists of a local limestone aggregate and is of low Ni value. The submission of coarse blank acts to test the efficacy of the sample preparation (crushing), determining if there is any sample contamination of this process. The fine blank (OREAS_22p) is used to measure background levels of Ni, Cu or Co in the laboratory analysis.

11.4.3 Check Assays

During the Stage 2 drilling program a duplicate check assaying program was initiated to compare initial Genalysis results with an independent external laboratory. ALS Chemex based in Perth were used as the independent check laboratory.

11.5 QA-QC

11.5.1 Overview

The acquisition of data that provide measures of analytical accuracy, sample representivity, sub-sampling quality and sample preparation quality are essential to determine the validity of an assay data set to be used for resource estimation.

Various measures are commonly used and include:

- Insertion of blind assay standards, of known grade, into the sample stream. Standards are used to assess the accuracy of the analytical data.
- Collection of duplicate samples, either identically re-split drill cuttings or re-sampling of remaining diamond core. Duplicate samples can be used to detect analytical error caused by the method used and care taken in sample collection.
- Insertion of coarse blank materials. These samples are subjected to the same sample preparation and can be used to detect poor hygiene issues, i.e. cross-contamination, during sample preparation.
- Repeat assaying of replicate samples from same sample pulps. These data provide a measure of the analytical precision achieved by the laboratory. This data is usually acquired as part of the normal service provided by the laboratory.
- Repeat or check assays determined at a different analytical laboratory (ALS-Chemex, Brisbane). This can be used to detect laboratory bias.
- Sizing analysis is used to evaluate the quality of the pulverising stages of sample preparation.

At the commencement of the 2004 drilling program at Ban Phuc the following measures were implemented:

- The use of assay standards;
- Check assays from a second laboratory;

- Sizing analysis; and
- Coarse and fine blanks.

11.6 Assay Standards and Blanks

Five different standards have been used, two low grade Nickel, one medium grade and two high grade. Details of material type, source and accepted grades are shown in Table 16 below. In addition two blank materials were also used, a quartz pulp and a coarse gravel.

Standards were inserted into the sample stream at a rate of one assay standard, one blank pulp, and one coarse blank for every 22 samples, to comprise a total batch of 25 items.

Table 16. Assay Standards and Blank Materials

Standard Name	Material Description	Determinations	Accepted Values		
			Ni%	Cu ppm	Co ppm
G_BM64	LG Nickel : Gannet Maggie	58	0.63	330	24
G_MHB1	LG Nickel : Gannet Maggie Hays B1	111	0.86	371	253
OREAS_14p	MED Nickel : Ore Research	72	2.09	1,000	751
OREAS_M3	HG Nickel : Ore Research Miitel	7	4.16	7,552	755
G_M4	HG Nickel : Gannet Miitel 4	27	4.55	4,151	864
OREAS_22p	BLANK : Ore Research qtz-iron	147	0.0001	0.0001	0.0001
Blank	Coarse Blank - River Gravel	226	0.0001	0.0001	0.0001

11.6.1 Analysis of QA-QC Data

11.6.1.1 Assay Precision

A total of 275 standards have been assayed and 373 blanks. Data is presented as 'control charts' plotting assay batch (essentially time) against each reported value. Also shown are the accepted value, the mean of the reported values and control limits set at +/- 2 standard deviations (s.d.) of the assayed mean. Comparison between the mean of the reported values and the accepted values shows how each standard has performed against expectation. Outliers, that can adversely affect the mean, have been removed so the calculated mean is a good indication of the laboratory's ability to replicate the accepted grade of the standard (Table 17). The 2 s.d. control limits are used to determine whether individual results are within acceptable deviation of the mean as determined by the particular laboratory.

11.6.1.2 Assay Bias

The results show that in general Ni and Co appear to be slightly under-reported (0 – 5%) by Genalysis whilst Cu is similarly over-reported, when the means are compared to the accepted value. Relative standard deviations in the order of 4-6% are good, which indicates the laboratory's internal repeatability for each standard is good (Figure 9 and Figure 10). The graphical representations show no concerning trends. There is some evidence of analytical drift in the data, particularly for standard G_MHB1, where short trends of consistently increasing grade are seen which are then terminated by an abrupt drop in grade followed by a repeat of the trend. The results though still remain within acceptable limits.

Table 17. Comparison of Assay means to Accepted Standard Values.

		Standard				
		oreas_m3	**g_m4**	**oreas_14p**	**g_mhb1**	**g_bm64**
Ni%	accepted value	4.16	4.55	2.09	0.86	0.63
	assayed mean	4.04	4.30	2.01	0.84	0.63
	%difference	-2.8%	-5.4%	-3.8%	-3.0%	0.6%
	Relative S.D.	8.0%	4.0%	4.0%	4.0%	4.0%
Cu ppm	accepted value	7,552	4,151	9,970	371	328
	assayed mean	7,366	4,260	9,811	383	345
	%difference	-2.5%	2.6%	-1.6%	3.3%	5.2%
	Relative S.D.	6.0%	3.0%	5.0%	4.0%	4.0%
Co ppm	accepted value	755	864	751	253	24
	assayed mean	730	862	704	255	23
	%difference	-3.3%	-0.3%	-6.3%	0.8%	-5.2%
	Relative S.D.	5.0%	2.0%	4.0%	5.0%	5.0%

11.6.1.3 Blanks

Blank results (Figure 9) show no serious indications of systematic cross-contamination due to poor laboratory hygiene. The barren quartz pulp has a mean assay value of 0.0025% Ni and shows only occasional low level spikes up to 0.01% Ni in about 2% of samples. This blank pulp also acts as an additional assay standard of very low grade. Clearly the level of over-reporting seen with this material is inconsequential.

The coarse blank samples report consistent results. Five results of 0.05% - 0.1% Ni, and elevated Cu and Co, are evident (less than 2% of the data) and one sample reported a value of >0.26% Ni, 1110 ppm Cu and 87 ppm Co.

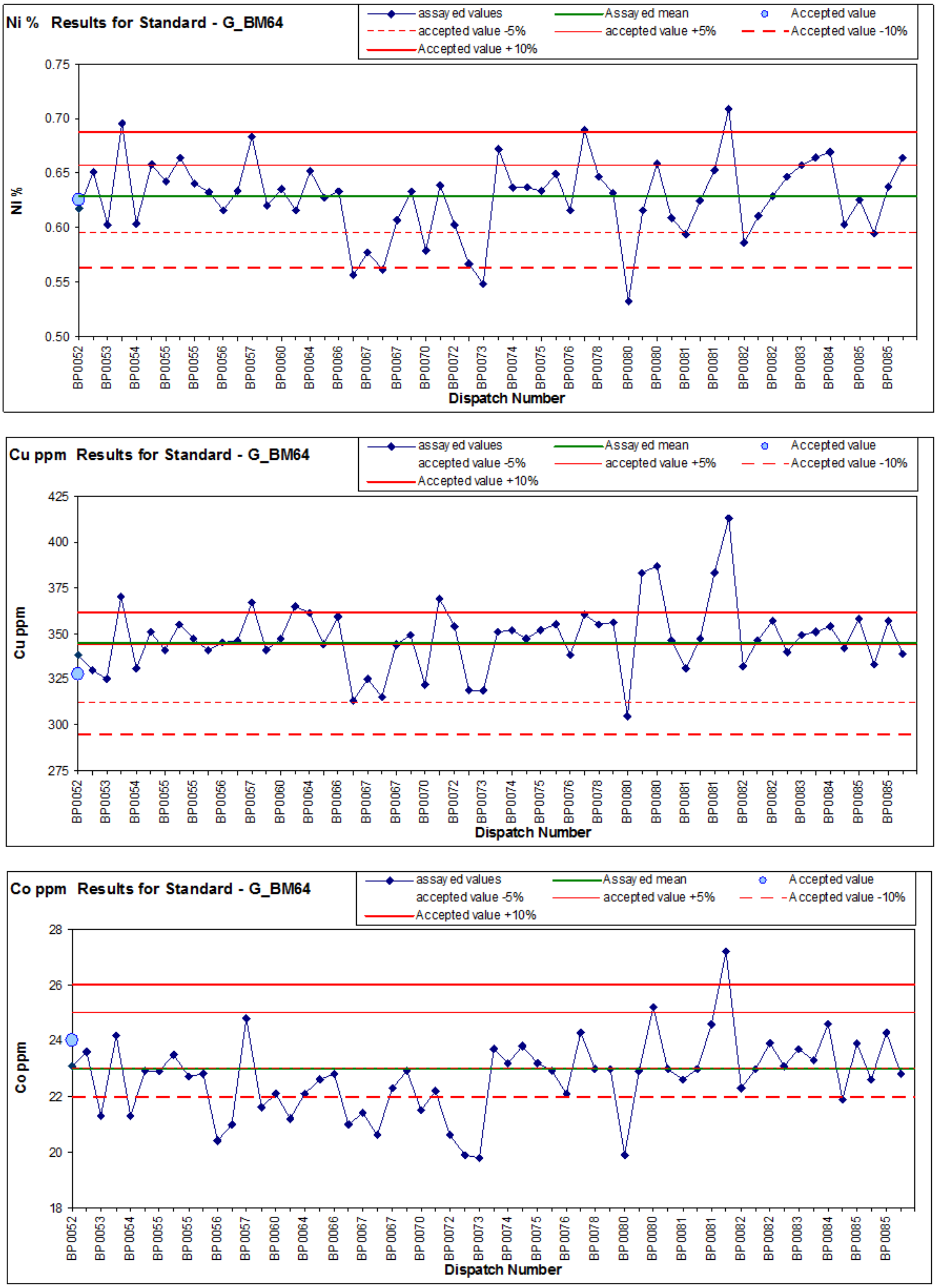


Figure 9. Results for Standard G_BM64

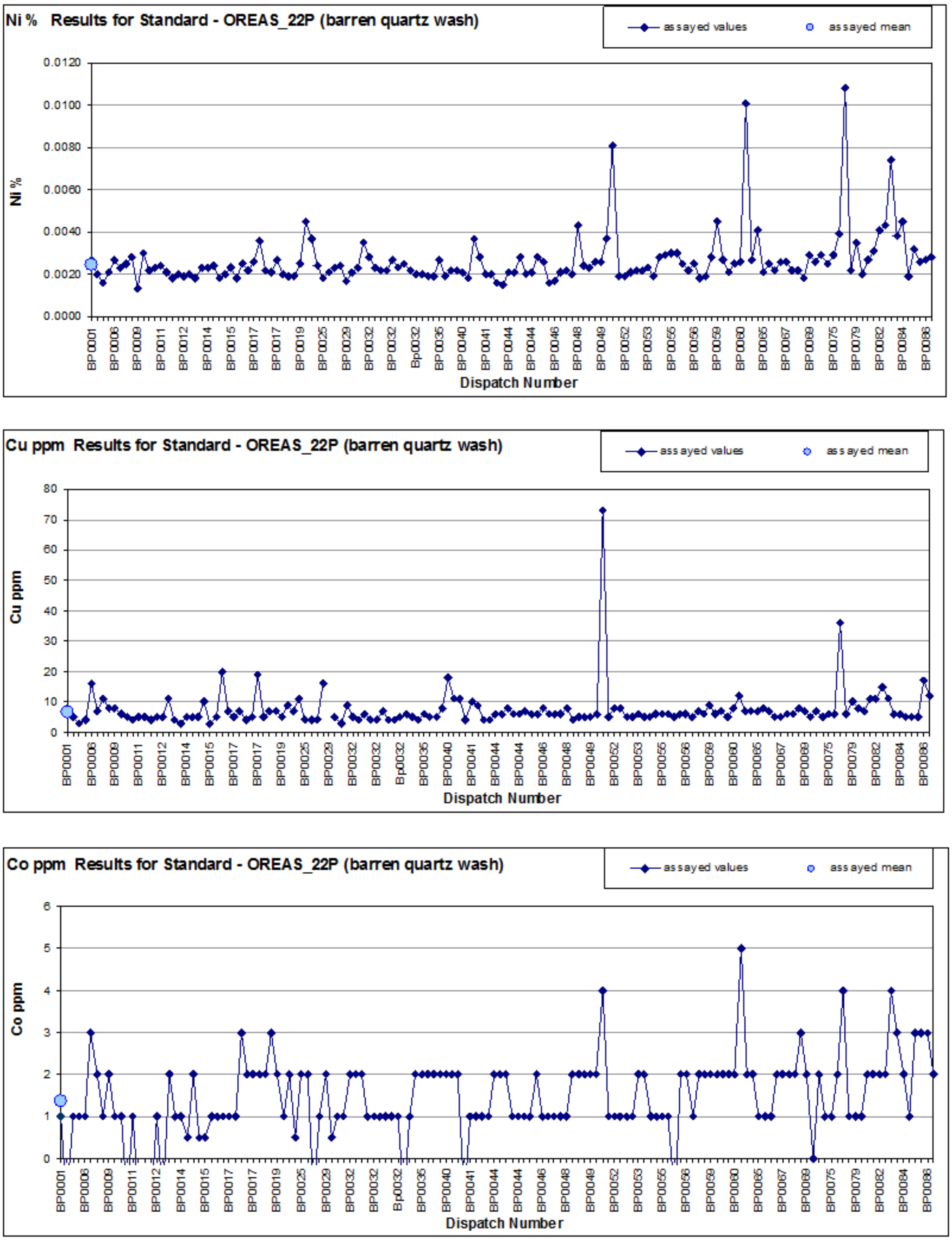


Figure 10. Assay Results for Blank Pulp OREAS_22P.

11.7 Sizing Analysis

Part of the sample preparation protocol employed by Genalysis was to fine-pulverize pulps so that 85% of material passes through a -75um mesh. This is to ensure adequacy of the grind and thereby ensuring efficient sample digestion. Excessive coarse material will reduce the effectiveness of the acid digest resulting in less than complete dissolution of Ni, Cu and Co. The laboratory randomly screened a number of samples from each batch to ensure this requirement was fulfilled.

Data is analysed by simply plotting the reported value of % passing 75um against dispatch number (Figure 11). The acceptable limit is also plotted. It is quite evident from Figure 11 that a large number of data do not pass the 85% limit.

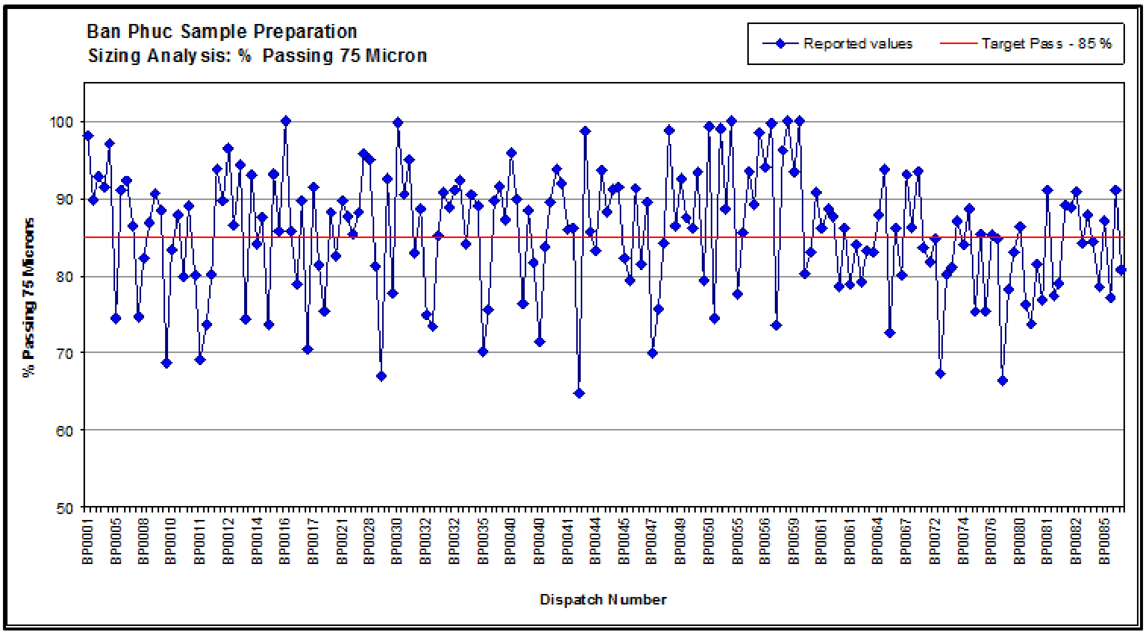


Figure 11. Sizing Analysis

There appears to be a point, at dispatch DPO60, after which there was deterioration in the completeness of the grind. This is confirmed by the data in Table 18. For DPOs 1 - 59 34% of samples tested failed, whilst from DPO 60 onwards this proportion increased to 60% failing.

Table 18. Sizing Data Analysis

	All Data	DPO 1-59	DPO 60+
Number of values	184	127	57
Mean %	85.6	86.7	83.1
Number <85%	77	43	34
% not passing	42%	34%	60%

It is recommended that a selection of samples from the low-performing batches be re-assayed following a regrind to assess the effect of the poor grinding.

11.8 3 vs 4-Acid Digest

The majority of the Ban Phuc assay dataset prior to 2004 was obtained by using a 3-acid digest with OES finish. It was decided to move to a 4-acid digest to obtain results that are more comparable to metallurgical head grades that are based on "total" techniques.

A series of check 3-acid vs 4-acid digest assays were completed in 2004 to ascertain the difference between the two assaying techniques. The grade variation between the two techniques (above 3% Ni) suggests that the grade of resource estimates of the MSV may be slightly under-estimated. The results between the two techniques at lower grades suggest that there is no significant difference.

11.9 ICP-OES vs Leco Sulphur

A large proportion of elevated Ni results in the upper oxidized and weathered portions of the Ban Phuc deposit show low sulphur values. Accordingly, sulphur duplicate work was carried out on a suite of 70 Ni enriched low sulphide (by ICP-OES) samples to ensure that the sulphur content was being reported correctly.

The results of the sulphur check assays show that up to approximately 200 ppm S, the Leco results are biased high with better agreement at higher grades.

11.10 Check Assays

During the Stage 2 drilling program a duplicate check assaying program was initiated to compare initial Genalysis results with an independent external laboratory. ALS Chemex based in Perth were used as the independent check lab.

Genalysis were requested to send 1 in every 10 sample pulps through to ALS for duplicate test work. The correlation between the Ni, Cu and S data sets was very good with the Pearson and Spearman correlation coefficients for the three datasets being very high.

11.11 Drill Hole Surveys

All drill holes have down hole surveys. These were completed at the time of drilling, however a number of holes from the BP series have been check surveyed by Surtron. A total of 62 holes (or 35%) were re-surveyed and the latter check survey results have been used preferentially in place of the original surveys.

A comparison of the results from the two surveys shows that there is a deviation between hole trajectories surveyed by each method. There does not appear to be any consistent relationship shown, however, with some check surveys being less inclined whilst others are steeper. The actual separation between holes obviously increases with depth, with a maximum distance of about 15 m in the deeper holes. There does not appear to be as much deviation in plan view. The error is not consistent with some deep holes showing very little deviation.

None of the older LK series holes were check surveyed. A comparison of the hole trajectories between BP04 series holes and LK series holes shows that the former commonly have shallower dips towards the bottom dips towards the bottom of the holes. This is seen in both shallower holes as well as deeper ones. These differences cannot necessarily be attributed to survey error as none of the LK series holes has been check surveyed.

11.12 Specific Gravity Analysis

Prior to the start of the 2004 drilling program, there was no routine collection of Specific Gravity data. Upon commencement of the Stage 1 program, a water immersion method was implemented at the Ban Phuc project to collect SG information for all core to be assayed. A retrospective program was run to gather SG information for available and suitable core drilled prior to 2004.

Specific gravities are determined by weighing individual pieces of core that represent the sampled interval. A subjective decision as to the process used for the determination of the rock's SG is made by the geologist logging the hole. This depends on the weathering, fracturing and apparent porosity.

An interval that is determined to be of a porous nature is:

- weighed in air (mass M);
- coated with wax (with specific gravity K);
- weighed in air with wax coating (mass P); and
- weighed in water with wax coating (mass S).

The following calculation is used to determine the SG of the interval:

- (SG_in_air)/((SGAirWax-SG_water_wax)-((SG_air_wax-SG_air)/SG_of_Wax))

An interval that is determined to be of a non-porous nature is:

- weighed in air (mass M); and
- weighed in water (mass W).

The following calculation is applied to determine the SG of the interval

- SGAir/(SGAir-SG_water)

11.13 Author's Opinion on Sample Preparation, Security and Analytical Procedures

The results show that in general Ni and Co appear to be slightly under-reported (0 – 5%) by Genalysis whilst Cu is similarly over-reported, when the mean values are compared to the accepted values. Relative standard deviations in the order of 4-6% are good, which indicates the laboratories internal repeatability for each standard is good. The graphical representations show no concerning trends. There is some evidence of analytical drift in the data, particularly for standard G_MHB1, where short trends of consistently increasing grade are seen which are then terminated by an abrupt drop in grade followed by a repeat of the trend. The results though still remain within acceptable limits.

CSA considers that the sampling preparation, security and analytical procedures were acceptable to industry standards.

12 Data Verification

12.1 Site Visit

Two of the authors from CSA (Bielin Shi and Gerry Fahey) were invited by BPNM to visit the site and to perform a preliminary review of the exploration property data at Ban Phuc Nickel deposit in northern Vietnam in June 2010. The purpose of this visit was to conduct an independent review on the geological control, mining geology conditions and field data collection as well as the established QA-QC procedures that were adopted on site.

The field work being carried out at the project was shown to be generally of a high standard with good attention to detail. CSA recommended some changes and additional work that could further enhance the quality of the data collected. As a final recommendation, CSA recommended that BPNM formalise the field work, sampling and assaying procedures and protocols into a manual to be used on site as a reference during subsequent field programs.

The validity of the database used for the Mineral Resource estimate of mineralization at the Ban Phuc deposit has been confirmed via checks for internal consistency and accuracy. As a result of these checks the authors consider that the drill hole data has been adequately validated with satisfactory data QA-QC analysis and is appropriate for use in the estimation of Measured, Indicated and Inferred Mineral Resources which are the subject of this technical report.

12.2 Data Validation

12.2.1 Database Validation

During preparation for estimation, the following validation steps were undertaken:

- Missing collar co-ordinates, hole depths, missing down hole surveys; miss matched collar, survey or assay depths; or over lapping intervals; and
- Missing or overlapping intervals for geology or assay interval data.

Primary assays fields were checked for missing assays, negative values and zero values (See Table 19).

- Negative assays which were determined to be below detection were replaced with a positive value of 0.001 %;
- Empty assays which were due to incomplete samples or missing core/chips were left as null samples. These will have no impact on interpolation, and the assumption is that the grade of these missing values is similar to that of neighbouring samples, and that local block interpolation will generate representative estimates based on neighbouring data contained in the search ellipse; and

- Zero grade values were replaced with nulls if determined to be true missing data, or a below detection positive value (0.001) otherwise.

Table 19. Summary of negatives in the database

Negative	Ni	Cu	Co	S	Mg	Fe	SG
-0.01	6	1	4		1	1	
-1.00E-04	1	127	6	8			
-0.01				36			
-0.005				8			
-0.001				40			
-0.016582					1		
missing	189	3,543	6,397	7,432	8,380	8,380	7,308

12.2.2 Data Summary

Original data files were exported from the CSA database as Micromine DAT files. Data manipulation and wireframe interpretation was carried out. Data was then exported to Datamine and Comma separated variable files. Datamine macros were used to import composite data and modelling parameters, followed by Macros using the ESTIMA process to estimate block grades using cut, uncut and flattening methods.

A summary of the Micromine database is shown below in Table 20.

Table 20. Ban Phuc MSV Ni database summary.

Database Name	Date Created	Database Type	Average Drilling Grid
Ban Phuc MSV	18-May-2010	Micromine DAT files	25m x 25m and 25m x 50m
File Name	Description		
BP _DBase_col.DAT	Collar data		
BP _DBase_sur.DAT	Downhole surveying data		
BP _DBase_ass.DAT	Sample assay data		
BP _DBase_geo.DAT	Geological and lithological data		

12.2.3 QA-QC

QA-QC processes were documented by H&S (2007) and BPNM (2008, 2010) for sampling and assaying. The results for Standards, Blanks and duplicates analysis are within the accuracy limits for these analytical techniques and, on the whole, show the quality of the analytical work to be satisfactory.

12.2.4 Data Sets

The original data was exported from the 'CSA_DBase' Access database. The data set was exported from Access and imported into Micromine and Datamine for the modelling process. Collar data as used in the resource estimation is illustrated below (See Figure 12).

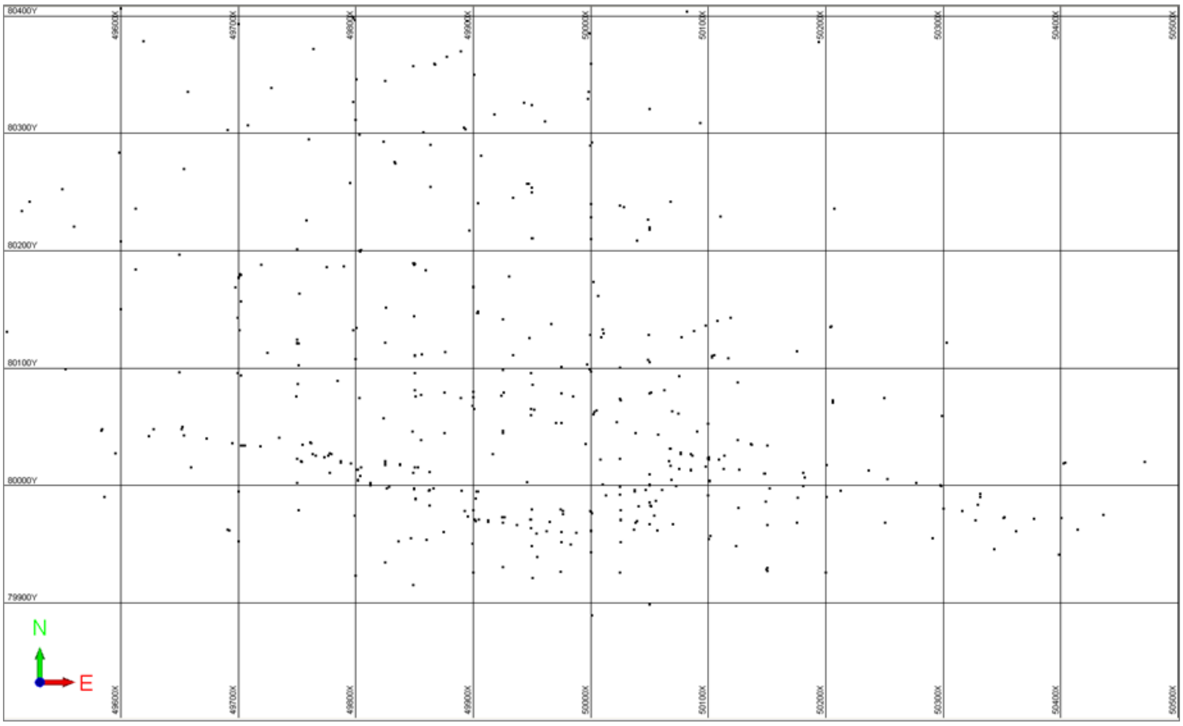


Figure 12. Plan view on Collar Location Data of Ban Phuc area

13 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Testwork Program

Phase 1 of the metallurgical testwork program was carried out prior to February 2005, with the results included in a preliminary assessment by Ausenco released in April 2005. This included comminution and preliminary flotation testwork on three samples from the deposit, two representing the DISS mineralization and one the MSV. Representivity of these samples was poor due to the limited number of drill intersections available at the time from previous drilling campaigns.

Phase 2 of the testwork program was conducted between March and August 2005 on the MSV only. Ausenco's preliminary assessment supported a change in the project scope to focus only on underground development of the MSV and work on the DISS mineralization was curtailed. Comminution and flotation testwork was completed on 17 composite samples representing different locations within the resource, a range of nickel head grades, and a range of nickel to copper ratios in feed. All the MSV intersections from the 2004 drilling program were used to make up these composites and, therefore, their representivity in terms of the total resource was high. Additional metallurgical tests were completed to cover other aspects of plant design and to meet the level of confidence required for a feasibility study. The results obtained from the metallurgical testwork on the MSV are outlined in the following sections.

Phases 1 and 2 were completed under the direction of Mr. Peter Lewis at Metcon Laboratories in Sydney and, in the case of the Phase 2 comminution testwork, at Ammtec Limited in Perth under the direction of Mr. Peter Lewis. Unless otherwise noted, the following information relates to the Phase 2 metallurgical testwork program. Phase 3 flotation and comminution testwork was conducted on a new composite of the MSV and a composite of the disseminated sulphide mineralization in 2008 under the direction of Metplant Engineering Services.

Work on the MSV composite focused on the reproducibility of the flotation performance obtained in the Phase 2 testwork program and validation of some process design parameters. The composite's aging characteristics were also evaluated.

Davis Tube sighter tests were conducted on the MSV composite and concentrate produced from the associated flotation tests to assess the potential upgrading achievable by magnetic separation of the pyrrhotite.

Flotation testwork conducted on the disseminated sulphide composite had the objective of producing a concentrate at acceptable grade and recovery levels, for further downstream processing.

Phase 4 flotation testwork was completed between April and September, 2011 under the direction of Mr. Steve Ennor on a sample of MSV to identify possible reagent and circuit configurations to produce separate copper and nickel flotation concentrates.

13.2 Comminution Testwork

A limited amount of comminution testwork was completed during Phase 1. The main and definitive comminution testwork was completed during Phase 2. The sample selection methodology and comminution results are detailed below.

13.2.1 Sample Selection

The Phase 1 program included comminution testwork on two samples representing tremolite altered dyke and unmineralized hornfels, which are the lithological units associated with the MSV. Bond rod and ball mill work indices and the abrasion index were determined for each sample. These results are included in Table 21. Clearly more comminution testwork was required to provide the necessary ore grindability and hardness information for comminution circuit design. The Phase 2 testwork program included determinations of Bond rod and ball mill work indices and abrasion indices, together with semi-autogenous grinding competency (SMC) testwork and unconfined compressive strength (UCS) testing. Samples from within and adjacent to the MSV were collected for this comminution testwork.

Sample selection was based on spatial location within the MSV, which was divided into 6 distinct zones for examination. Composite samples, which included internal waste and appropriate amounts of external waste, representing each of these zones were made up from all the MSV intercepts available from the 2004 drilling campaign. These zones represented the top, middle and bottom elevations from both the eastern and western sections of the deposit. These 6 depth composites were also subjected to flotation testwork.

It was recognized that due to mining dilution significant levels of unmineralized material adjacent to the MSV would be present in the mined ore, and the comminution characteristics of this dilution material were expected to dictate the design parameters for comminution circuit design. Consequently, although some external waste had already been included in the depth composites, composite samples representing only the waste material adjacent to the MSV were made up to correspond to each of the depth composites representing the MSV itself.

13.2.2 Comminution Results

Bond rod mill and ball mill work indices and the abrasion index were determined for each of the depth and waste composites. These results are summarized in Table 21 along with the results of the tests previously performed during Phase 1 on tremolite altered dyke and unmineralized hornfels.

The results indicated that there was relatively little variation in the rod and ball mill work indices within the MSV and the waste. However, they confirmed that the waste material was considerably harder than the corresponding MSV, with 20% to 30% higher work indices being reported.

The relatively high ratio between the rod and ball millwork indices indicated that scats generation within the grinding circuit might be significant. It also indicated that, if fully autogenous grinding (AG) or SAG milling were to be used, then recycle crushing would be required and the circulating load of pebbles was likely to be high. The abrasion indices of the waste material ranged between 0.10 and 0.25 placing this material in the medium to high category in terms of abrasiveness.

Table 21 Comminution Results Summary for Ore and Waste Composites

Ore composites							**Lab grind time for P_{80} (90µm)(min)**
Composite	**Description**	**Rod mill W*i***		**Ball mill W*i***		**Rod/Ball ratio**	
MSWT	Western section top	17.6	821	12.3	81	1.43	13.25
MSWM	Western section middle	18	822	12.7	81	1.42	13.5
MSWB	Western section bottom	17.7	837	12.7	82	1.39	13.75
MSET	Eastern section top	16.5	803	12.3	81	1.34	12.75
MSEM	Easter section middle	17.7	822	12.6	81	1.39	13.5
MSEB	Eastern section bottom	Insufficient sample to available complete tests 11.49					
	Minimum	16.5	803	12.3	81	1.34	11.5
	Maximum	18	837	12.7	82	1.42	13.75
	Average	17.5	821	12.5	81	1.4	13.04
Waste composites							**Lab grind time for P_{80} (90µm)(min)**
Composite	**Description**	**Rod mill W*i***		**Ball mill W*i***		**Rod/Ball ratio**	
MSWT	Western section top	21.1	764	15.1	77	1.4	0.2448
MSWM	Wester section middle	22.4	780	15.6	77	1.44	0.2444
MSWB	Western section bottom	21	831	15.4	72	1.36	0.2271
MSET	Eastern section top	20.9	807	15.3	80	1.37	0.1568
MSEM	Easter section middle	21.9	726	17.5	89	1.25	0.2294
MSEB	Eastern section bottom	18.5	782	13.3	74	1.39	0.0979
Phase 1	Tremolitized dyke	21	800	12.7	73	1.65	0.0344
Phase 1	Hornfels (sediments)	22.2	770	14.5	77	1.53	0.1756
	Minimum	18.5	726	12.7	72	1.46	0.03
	Maximum	22.4	831	17.5	89	1.28	0.24
	Average	21.1	783	14.9	77	1.42	0.18

UCS testwork was conducted on the waste composites and the results are shown in Table 22. A very wide range of results was obtained, with the most competent material generally occurring in the western section of the deposit

Table 22 UCS Results Summary (Waste Composites)

Area	Interval	UCS (MPa)	
Western section top	BP04-15 : 37.00-38.20 m	16.8	
Western section top	BP04-12 : 93.45-64.60m	43.9	
Western section top	BP04-15 : 27.80-29.00m	47.9	min
Western section top	BP04-12 : 84.40–85.65m	41.5	ma
Western section top	BP04-20 : 45.00-46.20m	117.6	ave
Western section middle	BP04-17 : 77.00-77.70m	190.6	
Western section middle	BP04-11 : 112.70-114.00m	18.3	
Western section middle	BP04-62 : 179.00-179.70m	33.9	min
Western section middle	BP04-17 : 99.30-100.00m	202.1	ma
Western section middle	BP04-65 : 203.00-203.70m	36.3	ave
Western section bottom	BP04-49 : 238.80-239.80m	32.7	
Western section bottom	BP04-49 : 224.60-225.60m	58.4	
Western section bottom	BP04-53 : 280.40-281.40m	24.3	min
Western section bottom	BP04-53 : 285.45-286.50m	17.9	max
Western section bottom	BP04-51A : 283.30-285.00	92.3	ave
Eastern section top	BP04-06 : 58.10-59.10m	25.7	
Eastern section top	BP04-09 : 45.00-46.00m	8.6	
Eastern section top	BP04-08 : 27.00-28.00m	93.8	min
Eastern section top	BP04-05 : 22.00-23.00m	91.9	max
Eastern section top	BP04-05 : 25.50-26.50m	80.5	ave
Eastern section middle	BP04-01A: 162.00-163.60m	13.6	
Eastern section middle	BP04-40 : 129.20-130.00.	32.8	
Eastern section middle	BP04-07 : 105.20-106.00m	78.9	min
Eastern section middle	BP04-29 : 77.00-77.80m	40.7	max
Eastern section middle	BP04-60 : 169.20-170.00m	20.3	ave
Eastern section bottom	BP04-63 : 261.75-264.00m	18.4	
Eastern section bottom	BP04-63 : 261.75-264.00m	9.8	
Eastern section bottom	BP04-63 : 261.75-264.00m	10.5	min
Eastern section bottom	BP04-43 : 293.45-296.45m	47.1	max
Eastern section bottom	BP04-43 : 293.45-296.45m	58	ave
	Minimum	8.6	
	Maximum	202.1	
	Average	53.5	
	Standard deviation	48.5	

SMC testwork was also carried out on the six waste composites to assess their variability in competency. A summary of the derived values determined from these tests is given in Table 23.The results showed that the waste material is in the upper 25% to 40% of samples in terms of hardness and impact breakage resistance (compared to the JKTech database). This provided a further indication of the potential difficulties with AG or SAG milling and the need for recycle crushing if AG or SAG milling were to be used.

Table 23. Derived Values for A*b and t10 (after JKTech)

Sample Designation	A*b				t10 @ 1kWh/t			
	Value	Category	Rank	%	Value	Category	Rank	%
Eastern section	46.41	medium	745	47.5	32.2	medium	777	49.6
Eastern section	41.04	mod. hard	585	37.3	29.4	mod. hard	589	37.6
Eastern section	31	hard	231	14.7	26.7	hard	410	26.1
Western section	33.65	hard	332	21.2	26.8	hard	426	27.2
Western section	35.47	hard	395	25.2	28.2	mod. hard	514	32.8
Western section	32.53	hard	295	18.8	26.9	hard	432	27.6
Rank refers to position in the JK Tech database from 1 (hardest) to 1,567 (softest)								
% refers to percentile in the JK Tech database from 1 (hardest) to 100 (softest)								

The Phase 3 testwork program included the determination of the Bond ball mill work index on the disseminated sulphides composite at 24.7kWh/t, which was significantly higher than the work indices previously obtained on MSV.

13.3 Flotation Testwork

13.3.1 Introduction

Based on marketing advice, the objective of all the flotation testwork was to produce a concentrate containing greater than 9% Ni and less than 5% MgO. No criteria were set for copper or other payable or penalty elements as these were expected to be of minor value or detriment.

All flotation testwork was completed at Metcon Laboratories in Sydney, Australia under the direction of Peter Lewis. Additional technical advice was provided by metallurgical consultant Arthur Dunstan who had extensive experience of Ni/Cu flotation.

During the Phase 1 program preliminary flotation testwork was completed on two samples of DISS mineralization and one sample of the MSV. Significant processing issues associated with the DISS mineralization were identified including high viscosity at typical flotation feed pulp densities, low nickel recoveries, high MgO in concentrates and high variability in results. The flotation results obtained on the sample of MSV indicated that a saleable concentrate could be readily produced at high nickel and copper recoveries, and that further testwork was warranted. However, attempts to produce separate copper and nickel concentrates were unsuccessful.

The subsequent flotation testwork in Phase 2 program was carried out during 2005 on 17 composite samples of the MSV representing different locations within the resource, a range of nickel head grades, and a range of Ni:Cu ratios in feed. The objective of the testwork was to generate reliable flotation performance predictions across the MSV.

Phase 3 of the program was conducted at AMMTEC laboratories in Perth, Australia in 2008 under the direction of Metplant Engineering Services. Flotation testwork was completed on a new composite of MSV and a composite of the disseminated sulphide mineralization. This phase focussed on flotation flowsheet validation and the aging characteristics of MSV ore. Further testwork was conducted on a disseminated composite to increase the level of confidence in the metallurgical behaviour of the ore type.

Summaries of the testwork results and performance predictions are given in the following sections.

13.3.2 Sample Selection Methodology for Phase 2 Flotation Testwork

Three sets of composite samples were prepared using half drill core from all the MSV intercepted by the 2004 drilling program, comprising a total of 211 m of core from 46 intercepts in 31 drillholes. They represented the following three factors that were considered most likely to result in variations in flotation performance and were:

- Six depth composites representing the top, middle and bottom elevations in both the eastern and western sections of the deposit. The objective of the testwork on these composites was to establish any variations in flotation performance with depth and along strike.
- Six grade composites representing a range of nickel head grades. The objective of the testwork on these composites was to examine the relationship between the nickel head grade and flotation performance.
- Five ratio composites representing a range of Ni:Cu ratios in feed. The objective of the testwork on these composites was to establish the effect of variations in the Ni:Cu ratio in feed on the grade of nickel and copper in the concentrates produced.

The composites included all internal waste within each MSV intersection and appropriate amounts of external waste so that they would be representative of expected mill feed. The description of the composites and their identification codes are shown in Table 24.

Table 24. MSV Composite Samples

Depth Composites	
MSWT	Western section top
MSWM	Western section middle
MSWB	Western section bottom
MSET	Eastern section top
MSEM	Eastern section middle
MSEB	Eastern section bottom
Grade Composites	
GC-3.5	4.32% Ni
GC-3.0	3.58% Ni
GC-2.5	3.09% Ni
GC-2.0	2.63% Ni
GC-1.5	2.12% Ni
GC-1.0	1.40% Ni
Ratio Composites	
RC-3.5	Ni:Cu ratio
RC-3.0	Ni:Cu ratio
RC-2.5	Ni:Cu ratio
RC-2.0	Ni:Cu ratio
RC-1.5	Ni:Cu ratio

13.3.3 Phase 2 Flotation Testwork Program

All composites were subjected to a standard set of flotation conditions, which had been established for the MSV during the Phase 1 flotation testwork program. The same reagents and pH were used in all cases, with the only variable being the addition rate of the sodium ethyl xanthate ("SEX") collector. The initial tests on all the composites were with a total addition rate of 75 g/t SEX to the rougher flotation stage. Subsequent tests were completed on most composites at higher or lower addition rates, with the variations in addition rates broadly in line with the nickel head grade. The nickel and copper recoveries obtained at nickel concentrate grades of 9.0%, 9.5% and 10.0% were calculated for each of the tests to provide a basis for comparing all the test results.

After comparing the results of all the tests completed on each composite, the results to be used in the prediction of plant performance were selected.

The flotation testwork was completed over a period of approximately three months and during this time it became clear that, despite storage of the composites under freezing conditions, some oxidation of the samples was occurring. This was reflected in the reduction with time of pentlandite/pyrrhotite selectivity during the initial stages of cleaning, which was confirmed by completing a number of duplicate tests later in the testwork program under identical conditions to earlier tests.

Locked cycle tests with recycling of the cleaner tailing to the head of the roughers were completed on five of the grade composites at the end of the main testwork program. The effect of sample aging was evident from the results obtained. In all cases but one much lower concentrate grades were obtained in the first flotation stage (i.e. without any recycle of cleaner tails) than had been obtained in the equivalent batch flotation tests completed earlier. In five out of 6 locked cycle tests, the final concentrate grades after 6 cycles were well below 9% Ni. It is not clear to what extent sample oxidation or the recycle of the cleaner tailings impacted on the low concentrate grades obtained – possibly both contributed. However, the locked cycle tests did demonstrate that recycling of the cleaner tailing was likely to result in only a negligible increase in nickel recovery.

Two additional bulk composites were made up to generate samples for supplementary testwork. One was made up from the ratio composites and selected drill intercepts to give a head grade close to the average nickel and copper head grades expected in practice. This was used to produce concentrate for detailed analyses and concentrate samples for marketing purposes and to provide concentrate and tailings samples for thickening, filtration and other testwork. The second composite was made up from the depth composites to provide a sample for testing an alternate flowsheet with open circuit cleaning, although this testwork was not completed.

13.3.4 Phase 2 Flotation Results

This section presents a summary of the results obtained in the Phase 2 flotation testwork program. The results shown are those that were used to develop the model for predicting flotation performance.

13.3.4.1 Depth Composites

The selected results for the 6 depth composites are summarized in Table 25.

Table 25. Selected Results for Depth Composites

Composite	Calculated Head Grade			At 9.0% Ni conc grade		At 9.5% Ni conc grade		At 10.0% Ni conc grade	
	Ni (%)	Cu (%)	Ni/Cu ratio	% Ni rec	Tailing (% Ni)	% Ni rec	Tailing (% Ni)	% Ni rec	Tailing (% Ni)
MSWT	2.66	1.07	2.49	90.3	0.35	89.8	0.36	89.3	0.37
MSWM	2.63	0.99	2.66	89.3	0.38	88.8	0.39	88.3	0.4
MSWB	2.89	1.23	2.35	89.6	0.42	88.4	0.46	87.3	0.49
MSET	2.43	0.82	2.96	87.7	0.39	86.8	0.41	86.1	0.43
MSEM	2.76	1.09	2.53	90.2	0.38	89.8	0.38	89.3	0.39
MSEB	2.79	1.76	1.59	88.5	0.44	85.1	0.55	Not achieved	

There was a general trend of decreasing recovery and increasing tailing grade with depth at each of the three concentrate grades. This applied particularly in the eastern section of the deposit where a concentrate grade of 10% Ni was not achieved on composite MSEB. This trend was against expectations, since it was thought that the mineralization nearest to the overlying oxidized zone would be likely to produce the worst flotation performance. Mineralogical analysis completed on each of the composites showed that most of the pentlandite in the two deepest composites, MSWB and MSEB, occurred at a finer grain size than in the other four composites. This is the likely reason for the poorer flotation performance observed for these composites.

Nevertheless, the results indicated that all of the depth composites produced saleable concentrates with grades of 9% Ni or more at nickel recoveries in the range 85% to 90%.

Optimization of the grade/recovery relationship will depend on the terms offered by the selected smelter. As these terms had not been defined at the time of examining the flotation results, a concentrate grade of 9.5% Ni was selected as the basis for performance predictions.

13.3.4.2 Grade Composites

The selected results on the six grade composites are summarized in Table 26.

Table 26. Selected Results for Grade Composites

Composite	Calc head grade			At 9.0% Ni conc grade		At 9.5% Ni conc grade		At 10.0% Ni conc grade	
	Ni (%)	Cu (%)	Ni/Cu	% Ni	Tailing	% Ni	Tailing	% Ni	Tailing
			ratio	rec	(% Ni)	rec	(% Ni)	rec	(% Ni)
GC-3.5	4.27	1.43	2.99	92.4	0.575	91.6	0.606	90.1	0.684
GC-3.0	3.60	1.56	2.31	91.4	0.489	90.5	0.519	89.1	0.580
GC-2.5	3.10	1.46	2.12	89.7	0.460	88.8	0.488	87.2	0.543
GC-2.0	2.64	1.20	2.20	91.1	0.318	90.4	0.339	89.6	0.357
GC-1.5	2.17	0.86	2.52	89.7	0.285	88.8	0.303	88.1	0.318
GC-1.0	1.43	0.80	1.79	84.5	0.255	83.3	0.273	Not achieved	

Figure 13 shows the relationship between the nickel head and tailings grades for the grade composites at the three concentrate grades examined.

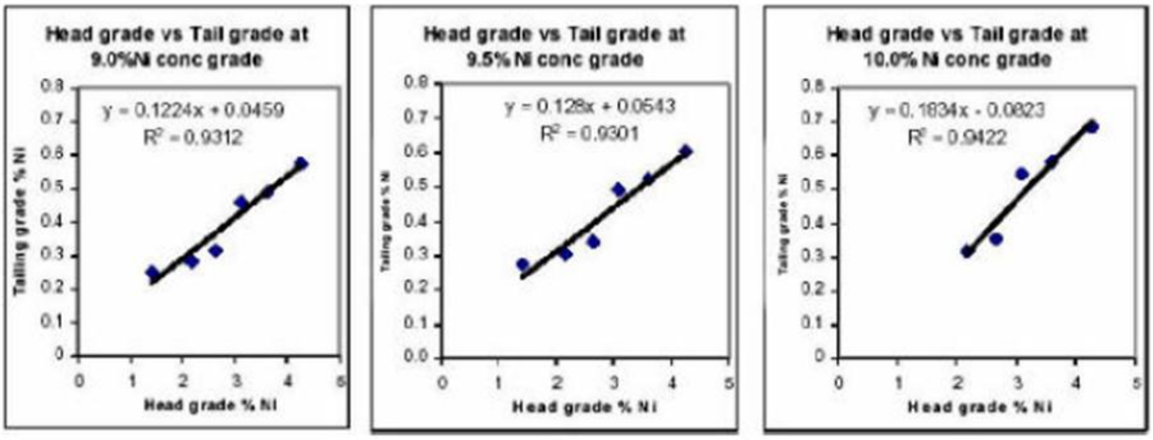


Figure 13. Nickel Head Grades vs Tailings Grades for Grade Composites

As was expected, at all three concentrate grades there was a strong correlation between the nickel head grade and tailings grade. Consequently, the relationship at 9.5% Ni concentrate grade was used as the basis for the flotation performance predictions.

13.3.4.3 Ratio Composites

The objective of the tests on the ratio composites was to establish the effect of the Ni:Cu ratio in feed on the concentrate grades. The selected results relevant to this objective are summarized in Table 27 followed by plots in Figure 14 of the relationship between the Ni:Cu ratio in feed against the Ni:Cu ratio in concentrate.

Table 27. Selected Results for Ratio Composites (Showing Cu Grades)

Composites	Calculated Head Grade			At 9.0% Ni conc grade		At 9.5% Ni conc grade		At 10% Ni conc grade	
	Ni	Cu	Ni/Cu	% Ni	Tailing	% Ni	Tailing	% Ni	Tailing
	(%)	(%)	ratio	rec	(% Ni)	rec	(% Ni)	rec	(% Ni)
RC-3.5	3.89	0.97	4.01	91.7	0.54	91	0.56	90.3	0.58
RC-3.0	3.82	1.18	3.24	91.2	0.55	90.7	0.56	89.9	0.59
RC-2.5	3.56	1.21	2.94	91.0	0.5	89.7	0.55	87.8	0.63
RC-2.0	3.25	1.36	2.39	88.2	0.56	86.5	0.62	85.1	0.67
RC-1.5	2.84	1.66	1.71	86.4	0.53	79.8	0.75	77.4	0.82

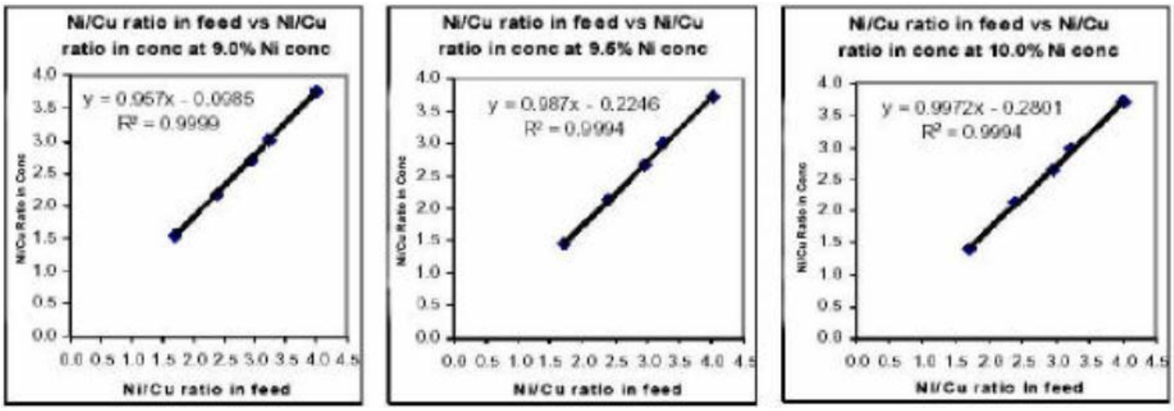


Figure 14. Ni:Cu Ratio in Feed vs Ni:Cu Ratio in Concentrate

There was an extremely strong relationship, approaching linear with a very high correlation co-efficient, between the Ni:Cu ratio in feed and the Ni:Cu ratio in concentrate at all three nickel concentrate grades. Therefore, for any given Ni:Cu ratio in feed the copper grade of the concentrate could be confidently predicted for concentrate grades of 9.0%, 9.5% and 10.0% Ni using these relationships.

Table 28 summarizes the nickel recoveries and tailing grades obtained on the ratio composites, followed by plots in Figure 15 showing the relationship between Ni:Cu ratio in feed and nickel tailing grade.

Table 28. Selected Results for Ratio Composites (Showing Ni Tailings Grades)

Composites	Calculated head grade			At 9.0% Ni conc grade		At 9.5% Ni conc grade		At 10.0% Ni conc grade	
	Ni	Cu	Ni/Cu	% Ni	Tailing	% Ni	Tailing	% Ni	Tailing
	(%)	(%)	ratio	rec	(% Ni)	rec	(% Ni)	rec	(% Ni)
RC-3.5	3.89	0.97	4.01	91.7	0.54	91.00	0.56	90.3	0.58
RC-3.0	3.82	1.18	3.24	91.2	0.55	90.70	0.56	89.9	0.59
RC-2.5	3.56	1.21	2.94	91.0	0.5	89.70	0.55	87.8	0.63
RC-2.0	3.25	1.36	2.39	88.2	0.56	86.50	0.62	85.1	0.67
RC-1.5	2.84	1.66	1.71	86.4	0.53	79.80	0.75	77.4	0.82

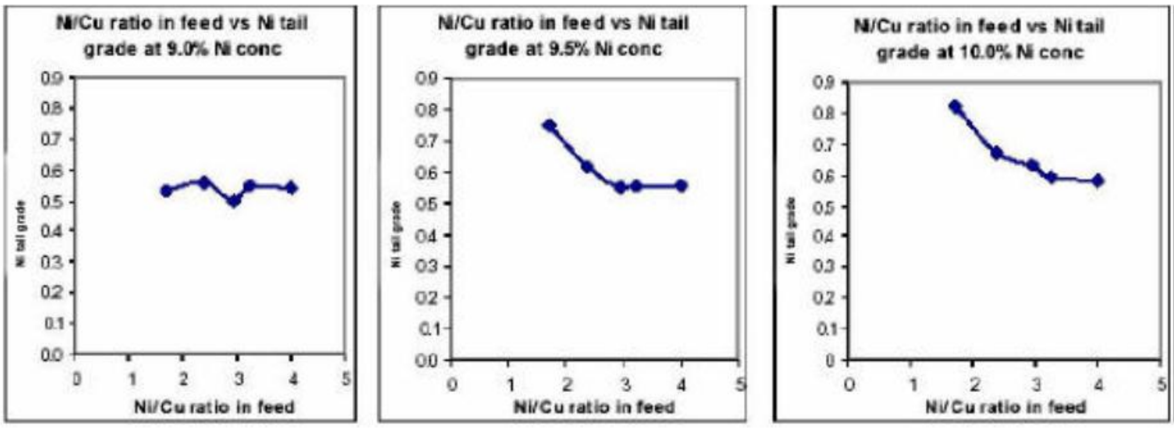


Figure 15. Ni:Cu ratio in Feed vs Nickel Tailing grade

The test data and plots indicated that nickel recovery decreased as the Ni:Cu ratio in feed decreased and as the target Ni concentrate grade increased. This effect can be explained by the observation that in all the tests completed the chalcopyrite floated more freely and rapidly than the pentlandite, such that the copper recoveries were consistently higher than the nickel recoveries. Therefore:

- as the amount of copper in feed increased the amount of copper diluting the concentrate would have increased, with the result that it became more difficult to achieve the target nickel concentrate grade without sacrificing some nickel recovery; and
- and this effect would have been exacerbated as the target concentrate grade increased beyond 9% Ni.

13.3.5 Lime as the pH Modifier

The Phase 1 flotation testwork had shown that slightly higher nickel recoveries were obtained with soda ash (Na_2CO_3) as the pH modifier than with hydrated lime ($Ca(OH)_2$). Consequently, soda ash was used as the pH modifier.

However, because a concentrate grade of 10% Ni had only been achieved in one test on composite GC 1.0 at a low nickel recovery, it was decided at the end of the main testwork program to assess whether improved flotation performance could be achieved on this composite using lime. Concentrate grades of 10% Ni at higher nickel recoveries were achieved in all the three tests completed with lime, which suggested that lime might be the better, and less expensive, pH modifier. However, as previously noted, the adverse effects of sample oxidation on flotation performance had become apparent by the end of the main testwork program, notwithstanding the samples having been kept frozen prior to testing. Therefore, as the tests with lime were the last to be completed it remained unclear whether lime was only superior to soda ash on partially oxidized mineralization. As no fresh samples remained, a re-assessment of the potential benefits of lime over soda ash could not be completed on fresh mineralization. Consequently, additional testwork on fresh samples comparing lime with soda ash was strongly recommended, but have not yet been completed.

13.4 Performance Predictions

Three factors were identified that affect the nickel recovery achieved at any given concentrate grade:

- the location within the deposit, with lower recoveries indicated at depth;
- the nickel head grade, with lower recoveries indicated as the head grade decreases; and
- the Ni:Cu ratio in feed, with lower recoveries indicated as the ratio decreases.

After eliminating the results from one composite, which were considered to be anomalous, the correlations developed between nickel tailing grade and both the nickel head grade and the Ni:Cu ratio in feed were used to predict the nickel recovery achieved at a concentrate grade of 9.5% Ni. A comparison of the actual results obtained on each of the composites and the predicted results is shown in Table 29 in which the predicted recoveries are shown in the penultimate column.

Table 29. Predicted and Actual Recoveries at 9.5% Ni Concentrate Grade

Composite	Head grade(% Ni)	Ni/Cu ratio	Actual Tailing (%Ni)	Actual Ni Recovery	Calculated on Ni head grade		Diff due to head grade	Effect of Ni/Cu ratio		Overall Diff in Ni recovery
					Tailing (%Ni)	Ni rec'y (%)		Change in Ni Rec'y	AdjNi rec'y	
MSWT	2.66	2.46	0.36	89.9	0.39	88.9	-1	-2.5	86.3	-3.6
MSWM	2.58	2.61	0.38	89	0.38	88.7	-0.3	-2.2	86.5	-2.5
MSWB	2.89	2.35	0.46	88.4	0.42	89.3	0.9	-2.7	86.6	-1.8
MSET	2.44	2.98	0.39	87.6	0.37	88.4	0.8	-1.5	86.9	-0.7
MSEM	2.77	2.5	0.35	90.7	0.41	89.1	-1.6	-2.5	86.6	-4.1
MSEB	2.79	1.59	0.55	85.1	0.41	89.1	4	-4.2	84.9	-0.2
Bulk	2.56	2.37	0.3	91.4	0.38	88.6	-2.8	-2.7	85.9	-5.5
CG-3.5	4.27	2.99	0.61	91.6	0.6	91.7	0.1	-1.5	90.2	-1.4
CG-3.0	3.6	2.31	0.52	90.5	0.52	90.6	0.1	-2.8	87.8	-2.7
CG-2.5	3.1	2.12	0.49	88.8	0.45	89.7	0.9	-3.2	86.5	-2.3
CG-2.0	2.64	2.2	0.34	90.4	0.39	88.8	-1.6	-3	85.8	-4.6
CG-1.5	2.17	2.52	0.3	88.8	0.33	87.8	-1	-2.4	85.4	-3.4
CG-1.0	1.43	1.79	0.27	83.3	0.24	85.5	2.2	-3.8	81.7	-1.6
Bulk	2.59	2.18	0.37	89.1	0.39	88.7	-0.4	-3.1	85.6	-3.5
RC-3.5	3.94	3.9	0.51	92	0.56	91.2	-0.8	0.3	91.5	-0.5
RC-3.0	3.92	3.16	0.52	91.7	0.56	91.2	-0.5	-1.1	90	-1.7
RC-2.5	3.56	2.94	0.55	89.7	0.51	90.5	0.8	-1.6	89	-0.7
RC-2.0	3.25	2.39	0.62	86.5	0.47	90	3.5	-2.7	87.3	0.8
RC-1.5	2.78	1.74	0.73	79.8	0.41	89.1	9.3	-3.9	85.2	5.4
Average	2.94	2.48	0.45	88.8	0.43	89.4	0.6	-2.5	86.9	-1.9

When deriving any predictions of plant performance some discount to the recoveries obtained in bench-scale laboratory testwork should be made to allow for the operating efficiencies that can and do occur in a full-scale plant operation. The average predicted nickel recovery in Table 29 is approximately 2% less than the actual nickel recoveries obtained in the flotation testwork, and this was considered to be a suitably conservative discount.

The procedure developed and used to predict the nickel recoveries in Table 29 and other plant performance data is described below.

Step 1 - Calculation of % nickel recovery

As noted previously, all predictions were based on the production of a 9.5% Ni concentrate. Here, the relationships between nickel head grade and nickel tailings grade (as shown in Figure 13) and between the Ni:Cu ratio in feed and the nickel tailing grade (as shown in Figure 14) were used to develop the formulae for predicting nickel recovery, which are as follows:

(1) t = (Nh x 0.128) + 0.0543

(2) Rp = (9.5 x (Nh – t) / (Nh x (9.5 - t)) x 100)

(3) A = (r x 1.9636) - 7.3501

(4) Rn = Rp - A

Where:

Nh = nickel head grade

t = nickel tailing grade

Rp = preliminary percent nickel recovery

r = Ni:Cu ratio in feed

A = reduction in preliminary nickel recovery due to Ni:Cu ratio in feed

Rn = final percent nickel recovery

Since the MSEB depth composite produced significantly lower recoveries than the other depth composites, a further reduction of 2% to the calculated final nickel recovery was applied to all mineralization deeper than 160 m below the base of oxidation in the eastern section of the deposit. The sub-division between the eastern and western sections of the deposit was at a point between sections 49,975 E and 50,000 E.

Step 2 – Calculation of the copper grade of the concentrate

Here the relationship between the Ni:Cu ratio in feed and Ni:Cu ratio in concentrate, as shown in Figure 14, was used. The formulae for predicting the copper grade of the concentrate are as follows:

[1] Cr = (0.987 x Fr) - 0.2246

[2] Cc = 9.5 / Cr

Where:

Fr = ratio of Ni:Cu in feed

Cr = ratio of Ni:Cu in concentrate

Cc = copper grade of concentrate

Step 3 – Calculation of percent copper recovery

This calculation was based on normal metallurgical balancing, the first step being to calculate the percent weight of concentrate. The formulae for calculating copper recovery are as follows:

[1] W = (Nh x Rn)/9.5

[2] Rc = (W x Cc)/Ch.

Where:

Nh = nickel head grade

Rn = final percent nickel recovery (from Step [1] above)

W = percent weight of concentrate

Ch = copper head grade

Cc = copper grade of concentrate (from Step [2] above)

Rc = percent copper recovery

Step 4 – Calculation of the Co grade of the concentrate

There was a very strong and consistent correlation between the nickel and cobalt head assays for all the 17 composites tested, with an average Ni:Co ratio in feed of 32.86. This indicated that the cobalt was in solid solution within the pentlandite and that the amount of cobalt in the concentrate would be directly related to the nickel assay.

Therefore, knowing the nickel grade of the concentrate, the cobalt grade can be calculated using the following formula:

(1) Co = C(Ni)/32.86

Where:

C(Ni) = nickel grade of concentrate

Co = cobalt grade of concentrate.

The ratio used was in close agreement with the average Ni:Co ratio of 31.49 for the seven concentrate samples on which detailed analyses were completed, as shown in Table 29.

It followed that cobalt recovery would be equal to the nickel recovery.

MgO assay of concentrate

In all the tests completed on the 17 composites the MgO assay of the final concentrate was well below the assumed 5% penalty level. Therefore, it was not necessary to develop a method of predicting the MgO assay of the concentrate.

13.5 Mineralogy

Mineralogical analysis was conducted on the MSV sample used in the Phase 1 testwork program. This showed that it contained large amounts of pyrrhotite and low levels of MgO bearing minerals. Nickel mineralization was shown to be dominated by pentlandite, which generally occurred as coarse, discrete grains or associated to varying degrees with pyrrhotite. It was concluded that the effectiveness of achieving selectivity against pyrrhotite during flotation would dictate the nickel concentrate grades and recoveries achieved.

For the Phase 2 program, the six depth composites were submitted for mineralogical examination by Roger Townend and Associates. In general terms, the results were comparable to the sample examined during Phase 1, with nickel occurring predominantly as pentlandite, either as discrete grains or associated with pyrrhotite. An excerpt from the Townend report follows and provides a link between the observed mineralogy and the poorer flotation performance of the deeper composites:

"*The six composite samples are broadly similar in mineralogy. The nickel mineral present is pentlandite, which is sometimes altered to violarite, and the copper mineral is chalcopyrite. They occur in two main lithologies: a tremolite / chlorite schist, and a graphite / feldspar / mica / quartz schist. Two of the composites (MSEB and MSWB) gave poorer nickel flotation results than the other composites. The only apparent difference between these two and the other four is the nature of the pentlandite. In MSWB and MSEB most of the pentlandite occurs as fines, with average grain sizes of 20-30 microns, in coarse pyrrhotite. Pentlandite in the other four composites is much coarser when in pyrrhotite, and discrete pentlandite may dominate.*

There is one qualification to this. In composite MSWM, there is also a significant amount of fine pentlandite in pyrrhotite, although discrete pentlandite is the dominant mode of occurrence."

13.6 Supplementary Metallurgical Testwork

The following section provides a summary of the supplementary testwork that was included in the Phase 2 program to provide design criteria for other sections of the plant. Unless otherwise indicated, the products used in the supplementary testwork were obtained from large scale laboratory flotation tests on a bulk composite sample made up from the ratio composites and some residual intercept samples to give nickel and copper head grades as close as possible to the expected average mill feed grades.

13.6.1 Thickening testwork

Laboratory scale thickening tests were conducted by Outokumpu Technology to define the performance of conventional thickening of the flotation concentrate and high-rate thickening of the tailings. The testwork included static and dynamic tests and examined the effects of a range of parameters including thickener feed density, settling rate, flocculant addition and underflow density.

Relatively high settling rates and underflow densities were indicated for both the tailings and concentrate. The key outcomes are summarized as follows:

- The optimum feed solids concentration for both the tailings and concentrate samples was approximately 15.0% w/w solids.
- Thickened concentrate densities of 70 - 80% w/w solids should be possible in practice.
- Suspended solids in the concentrate thickener overflow were expected to be less than 150 ppm, provided that excessive froth is not created in practice.
- Thickened tailings densities of 60 - 70% w/w solids should be readily achieved.
- Suspended solids in the tailings thickener overflow were expected to be less than 200 ppm.
- Based on a small number of screening tests, a low anionic charge flocculant was shown to be superior in terms of overflow clarity and underflow compaction.
- Solids loading rates of 0.5 - 1.2 t/m^2h for tailings and 0.25 t/m^2h for concentrate were achieved and should be used as the basis for thickener design.

13.6.2 Tailings slurry viscosity testwork

Slurry viscosity measurements were obtained for un-thickened (~30% w/w solids) and thickened (~60% w/w solids) tailings. In both cases the tests indicated low viscosities of less than 70 Cp at a shear rate of 100 sec-1. These observations enabled the tailings thickener underflow density to be confidently set at 60% w/w solids for slurry pumping system design.

13.6.3 Concentrate filtration testwork

Filtration tests on concentrate thickened to 70% w/w solids were carried out by Larox to determine the effectiveness of pressure filtration and to define the operating variables.

A bench scale pressure filter test rig was used, which enabled the impact of cake thickness, pressing time and air drying time to be examined. The key findings from the testwork can be summarized as:

- The material dewatered reasonably well with pressing times of less than one minute generally required.
- Cake moistures of less than 8.0% could be readily achieved.

- A cake thickness of 50 mm was considered to be the optimum, although low cake moistures could also be achieved with thinner cakes, albeit at a lower filtration capacity.
- Pressure filter capacities of 700 to 1,000 kg/m2h were indicated, depending on cake thickness and moisture targets.
- Visually clear filtrate was observed using the laboratory test cloth, and the cake discharged easily.
- Air drying was effective in removing the remaining moisture from pore spaces within the cake. Air consumption of 1.0 Nm3/m3.min should be applied for design at a supply pressure of ~ 8 bar.

In summary, the filtration results indicated that low filter cake moisture levels could be readily achieved at relatively high filtration rates.

13.6.4 Determination of concentrate Transportable Moisture Limit and potential for self-heating

Using the sample of filtered concentrate produced at Larox, the concentrate transportable moisture limit (TML) was determined to be 8.4%. The test was completed by Australian Testing Sampling & Inspection Services (ATSIS) using the flow table method. ATSIS also completed an autopyrolysis test on the same concentrate sample, which indicated that it would not be classified as self-heating in accordance with Class 4.2 of the International Maritime Dangerous Goods Code, 2002.

13.6.5 Detailed Concentrate Analysis

Products from the flotation tests on the 6 depth composites were combined to form concentrates with an estimated grade of 9.5% Ni. These were then subjected to detailed analyses for marketing purposes. A 5 kg flotation test was completed on the bulk composite to provide samples of concentrate to send to smelters interested in purchasing the concentrate. A sample of this concentrate was also subjected to detailed analyses. The detailed analyses of the seven concentrates are shown in Table 30.

Table 30. Detailed Analyses of Flotation Concentrates

Assay	Units	Concentrate Produced from Composites						
		MSWT	MSWM	MSWB	MSET	MSEM	MSEB	Bulk
Ni	%	9.43	9.7	9.42	9.39	9.59	9.29	10.4
Cu	%	4.16	3.97	4.7	3.6	4.09	6.1	5.35
Co	%	0.281	0.3	0.29	0.31	0.31	0.3	0.327
Pt	ppm	0.5	0.2	0.18	0.1	0.11	0.08	0.33
Pd	ppm	0.16	0.16	0.18	0.1	0.11	0.08	0.33
Au	ppm	0.24	0.17	0.18	0.18	0.14	0.12	0.17
Ag	ppm	5	6	8	6	6	6	7
Fe	%	47.3	43.2	44.9	38.2	42.8	43.3	40.3

Assay	Units	Concentrate Produced from Composites						
		MSWT	MSWM	MSWB	MSET	MSEM	MSEB	Bulk
S	%	36.7	31	33	28.5	31.4	32.9	33.4
MgO	%	0.44	0.75	0.17	2.54	0.75	0.33	0.91
Al2O3		0.35	1.07	0.95	1.7	1.47	0.71	1.13
SiO2	%	2.2	4.72	3.7	8.48	5.64	2.8	5.09
CaO	%	0.39	0.76	0.56	1.62	1.06	0.4	0.87
As	ppm	<10	<10	<10	<10	<10	<10	130
Bi	ppm	<10	<10	<10	<10	<10	<10	<10
Cd	ppm	114	111	121	103	117	126	108
Cl	ppm	200	300	<100	100	200	100	100
Cr	ppm	270	300	280	380	380	270	320
F	ppm	100	100	100	100	200	200	90
Hg	ppm	30	30	30	40	40	30	50
Mn	ppm	70	168	250	231	128	170	157
Mo	ppm	<5	<5	<5	<5	<5	<5	<5
Pb	ppm	<5	60	74	8	<5	34	68
Sb	ppm	<5	<5	<5	<5	<5	<5	<5
Se	ppm	35	40	20	40	10	15	20
Sn	ppm	<10	<10	10	<10	<10	<10	<5
Te	ppm	4.4	8.6	8	6.2	5.6	5.8	<10
Zn	ppm	240	195	247	349	186	209	202

The concentrate produced in the 5 kg flotation test was also subjected to size analysis. Its 80% passing size was 78 µm.

13.6.6 Tailings Physical and Geochemical Testwork

Two samples produced during the Phase 2 testwork program were sent to Knight Piésold in Perth for tailings consolidation testwork associated with the design of the tailings storage facility. The samples supplied were:

- Unthickened tailings at approximately 30% w/w solids, which had been produced in the laboratory flotation tests on the bulk composite.
- Thickened tailings at approximately 60% w/w solids, which had been produced by Outokumpu during the thickening testwork.

13.6.7 Acid Generating Potential Testwork

Three composite samples of flotation tailings were supplied for acid generating potential testwork. The composites were made up to represent the estimated minimum, average and maximum grades of sulphur that were expected in flotation tailings.

13.7 Phase 3 Metallurgical Testwork

This testwork was conducted by AMMTEC in Perth on a new composite of MSV and a composite of the disseminated sulphide mineralization.

13.7.1 Sample Selection

A sample of MSV was obtained from a drillhole which intersected 1.65m of massive sulphides at a depth 120.35 – 122.00m. This was combined with 0.7m of metasediments from the hanging wall and 0.7m of tremolite dyke from the foot wall to provide, as far as was practicable, a composite sample that was representative of mine ore/waste dilution.

The disseminated sulphide composite was retrieved from Adit 202-IVA to provide a broad representation of the disseminated mineralization lithology.

The composition of the new MSV composite and its head assays are given in Table 31 and the head assays of the disseminated sulphides composite are given in Table 32.

Table 31 ***MSV Composite and head assays***

Rock Type	Weight (%)	Ni (%)	Cu (%)	Fe (%)	MgO (%)	S (%)
Metasediment	19.6	0.47	0.51	6.09	1.39	2.76
Massive Sulphide	59.6	5.76	1.12	46.6	0.55	33.1
Tremolite Dyke	20.8	0.37	0.25	8.35	22.2	2.44
MSV composite	100.0	3.60	0.82	30.70	5.22	20.78

Table 32 Disseminated Sulphides Composite Head Assays

Rock Type	Ni (%)	Cu (%)	Fe (%)	MgO (%)	S (%)
Disseminated Sulphides	1.00	0.23	5.25	34.3	1.10

13.7.2 Flotation Testwork Results

Two flotation tests were completed on the new MSV composite. The first test failed to achieve the flotation performance consistently achieved in the Phase 2 testwork program. However, the soda ash addition and consequently pH were increased in the second test and significantly better flotation results were obtained, such that the weighted average results of the two tests were similar to those achieved in Phase 2. This is illustrated in Table 33.

Table 33 Comparison of Phase 2 and 3 MSV results

	Soda ash	pH	Calc head	Concentrate	
	(g/t)		% Ni	% Ni	% Ni Rec'y
Phase 3 Test 1	350	8.20	3.71	7.12	94.6
Phase 3 Test 2	1200	9.50	3.79	12.4	86.6
Phase 3 Average	775	8.85	3.75	8.97	90.6
Phase 2 result			3.60	9.00	91.4

It is known that some pyrrhotite can oxidise rapidly and adversely affect flotation performance. Therefore, tests were conducted on the MSV composite to examine the effect of aging on flotation performance. Tests were completed under exactly the same conditions after 0, 1, 2, 4 and 8 weeks of dry storage. A second series of tests were completed after the sample had been wetted each week to accelerate oxidation. Over the 8 week period of "dry" aging there was no obvious decrease in the nickel and copper recoveries, but from week 4 onwards there was an increase in weight recovery accompanied by a decrease in the nickel and copper concentrate grades.

For the first 4 weeks of "wet" aging there was a slight but steady decline in both nickel and copper concentrate grades without any apparent change in recoveries. However, after 8 weeks there was a dramatic reduction of 18% in the nickel recovery and 6% in copper recovery. This testwork reflected a 'worse-case' scenario as the samples in storage had been crushed to less than 3mm, thereby providing conditions which would be more prone to oxidation than would be the case for run-of mine (ROM) ore. However, the work did indicate that careful management of ROM pad stockpiles will be needed to minimize any potential oxidation of the sulphide minerals.

13.7.3 Magnetic Separation tests

Davis Tube magnetic separations to remove pyrrhotite were completed on the MSV composite and the flotation concentrate produced after 1 week of aging under dry conditions. Summaries of the results obtained are presented in Table 34 and

Table 35.

Table 34 Magnetic Separation - Concentrate

Product	Weight (g)	Weight (%)	Assays / Distributions					
			Nickel		Iron		Sulphur	
			%	% dist	%	% dist	%	% dist
Mags	8.6	43.5	12.7	29.7	46.0	50.7	35.1	45.0
Non-Mags	11.2	56.5	23.1	70.3	34.4	49.3	35.1	55.0
Total	19.9	100.0	18.6	100.0	39.4	100.0	34.0	100.0

Table 35 *Magnetic Separation - MSV*

Product	Weight (g)	Weight (%)	Assays / Distributions					
			Nickel		Iron		Sulphur	
			%	% dist	%	% dist	%	% dist
Mags	13.7	34.5	2.94	15.5	56.1	38.2	39.8	37.4
Non-Mags	26.0	65.5	8.43	84.5	61.8	61.8	35.2	62.6
Total	39.7	100.0	6.53	100.0	50.7	100.0	36.8	100.0

In both cases a high percentage of the nickel reported to the magnetic fraction, such that upgrading by magnetic separation would not be viable.

1.7.4. Flotation of Disseminated Sulphide Composite

Six tests were completed on this composite using a number of alternative methods to reject the large amount of magnesia minerals present. A summary of selected rougher and cleaner flotation results is presented in Table 36.

Table 36 Summary of Results Obtained on Disseminated Sulphide Composite

Depressant		Concentrate					
		Nickel		Copper		MgO	
Name	Dose (g/t)	%	% Dist	%	% Dist	%	% Dist
Guar	75	9.16	65.7	0.33	12.4	17.1	3.9
Soda Ash + MBS + CMC	1670/ 400/200	4.54	78.2	0.32	26.6	26.8	15.5
Sodium Silicate	1700	14.7	55.6	0.34	7.3	5.69	0.71
Sodium Silicate	3600	13.3	72.4	0.45	12.8	8.19	1.42
Soda Ash +MBS + CMC + Sodium Silicate	1000/ 400/ 200 / 700	15.1	67.4	0.41	10.4	7.60	1.17
Soda Ash +CMC + Sodium Silicate	1000/ 200 / 700	15.3	58.9	0.38	8.2	6.82	0.87

Encouraging concentrate grades of 13 to 15% Ni were obtained at nickel recoveries ranging from 55 to 72%. The composite appeared to be more amenable to a reagent regime involving dispersion rather than depression of gangue minerals, and the best result of 72.4% nickel recovery to a concentrate assaying 13.3% nickel and 8.2% MgO was achieved with the relatively high addition of 3,600g/t of sodium silicate.

Further testwork is required to optimise the flotation performance achievable on the disseminated mineralization. The possibility exists that a flotation concentrate produced from the disseminated mineralization could be blended with that produced from the MSV to give a combined concentrate that meets the required marketing specifications, particularly with respect to its magnesia content.

13.8 Phase 4 Metallurgical Testwork

A limited number of 6 tests were completed to assess the potential for producing separate nickel and copper concentrates. The testwork was completed on the sample of MSV that had been used in the Phase 3 testwork program, and which had been stored in a freezer to minimise oxidation. However, hanging wall and footwall dilution was not included, as was the case for the Phase 3 testwork. Consequently, as shown in Table 37, its nickel and copper head assays were much higher than any MSV samples previously tested.

Table 37 Head Assays of Phase 4 MSV sample

Ni (%)	Cu (%)	Fe (%)	MgO (%)
7.61	1.5	51.8	0.135

The testwork showed that differential flotation of the nickel and copper can be achieved. The best result obtained is summarized in Table 38. In this test bulk copper/nickel rougher flotation was followed by cleaner flotation at a high pH of 12 to depress the nickel sulphides, with the copper concentrate equating to the cleaner concentrate and the nickel concentrate to the cleaner tailing.

Table 38 Results of differential copper/nickel flotation

	Nickel		Copper	
	Assay (%)	Distribution (%)	Assay (%)	Distribution (%)
Copper concentrate	8.07	5.19	21.8	68.5
Nickel Concentrate	11.6	85.4	0.8	29.2

Further investigations would be required before any decision can be taken to produce separate copper and nickel concentrates rather than a combined nickel/copper concentrate. This would involve: optimisation of flotation conditions on a sample with more representative nickel and copper head grades; concentrate marketing studies; revision of the plant flowsheet including management of the water circuits which may be problematic; and revision of plant capital and operating costs.

14 Mineral Resource Estimates

14.1 Software

Three dimensional (3D) mineralization interpretations were carried out using Micromine software. The interpretations of the lodes at Ban Phuc MSV deposit were initially digitised as individual sections prior to being triangulated into 3D solids. Estimation of the resource was completed using Datamine v3.17 software.

14.2 Geological Interpretation

14.2.1 Interpretation

A total of 40 sections at 10m to 13m spacing were interpreted from 49,700m E to 50,250m E, covering the extent of the known mineralization in the Ban Phuc area. The interpretation and wireframes were generated based on a 25m × 25m and 25m × 50m exploration drilling patterns. The interpretation of the mineralization as Micromine strings on each lode has been summarized in the following sections.

14.2.2 Wireframe and Domaining

Wireframe solids were generated based on the sectional interpretations provided by AMR to delineate the lodes of Ni, Cu, Co, S, Fe and Mg mineralization. A lower cut-off of 0.4% Ni combined with the MSV and Disseminated (DISS) geological logging was used to define the mineralized envelopes.

The interpreted mineralized domains consist of two primary mineralization envelopes for MSV and three for DISS that are likely to be connected the extensional and infill drilling. Figure 16 to Figure 19 demonstrate the outlines of the modelled mineralized lodes.

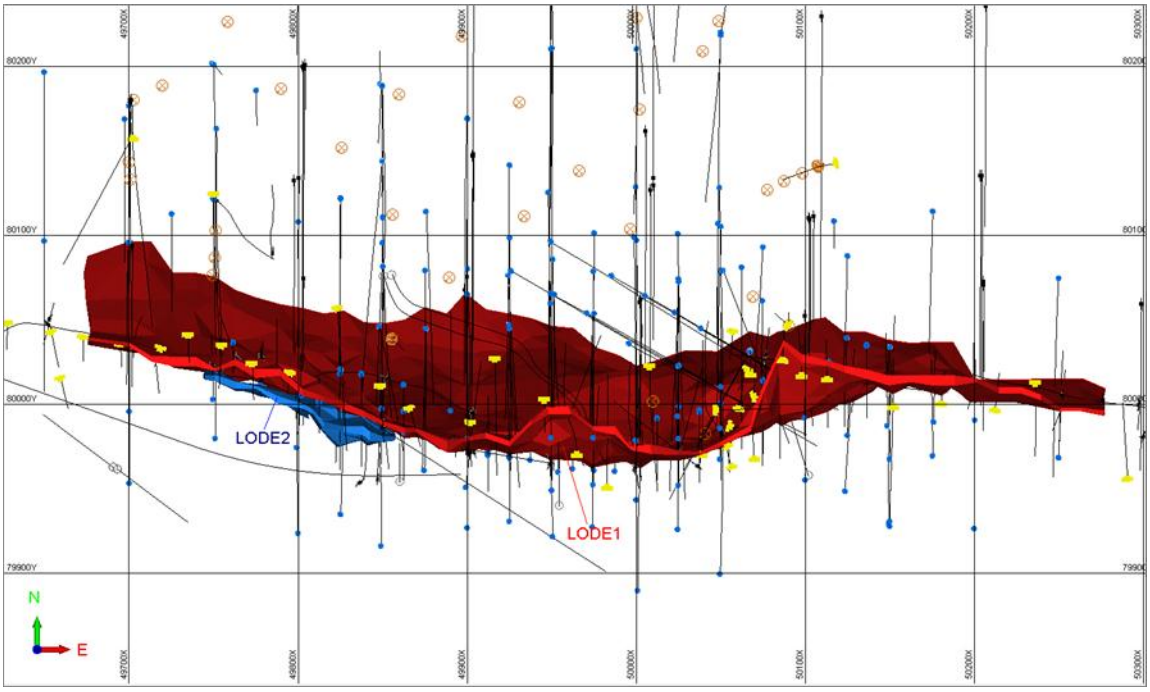


Figure 16. Plan view on extents of the modelled mineralized MSV lodes with drill holes collars

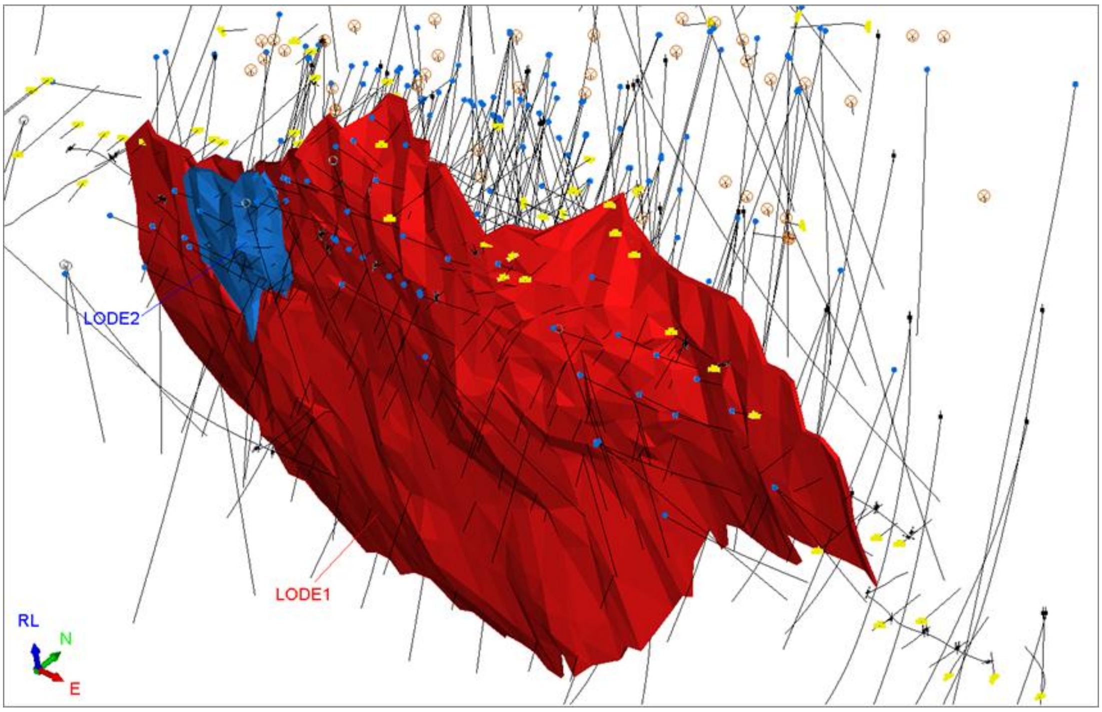


Figure 17. 3D view on extents of the modelled mineralized MSV lodes with drill holes collars

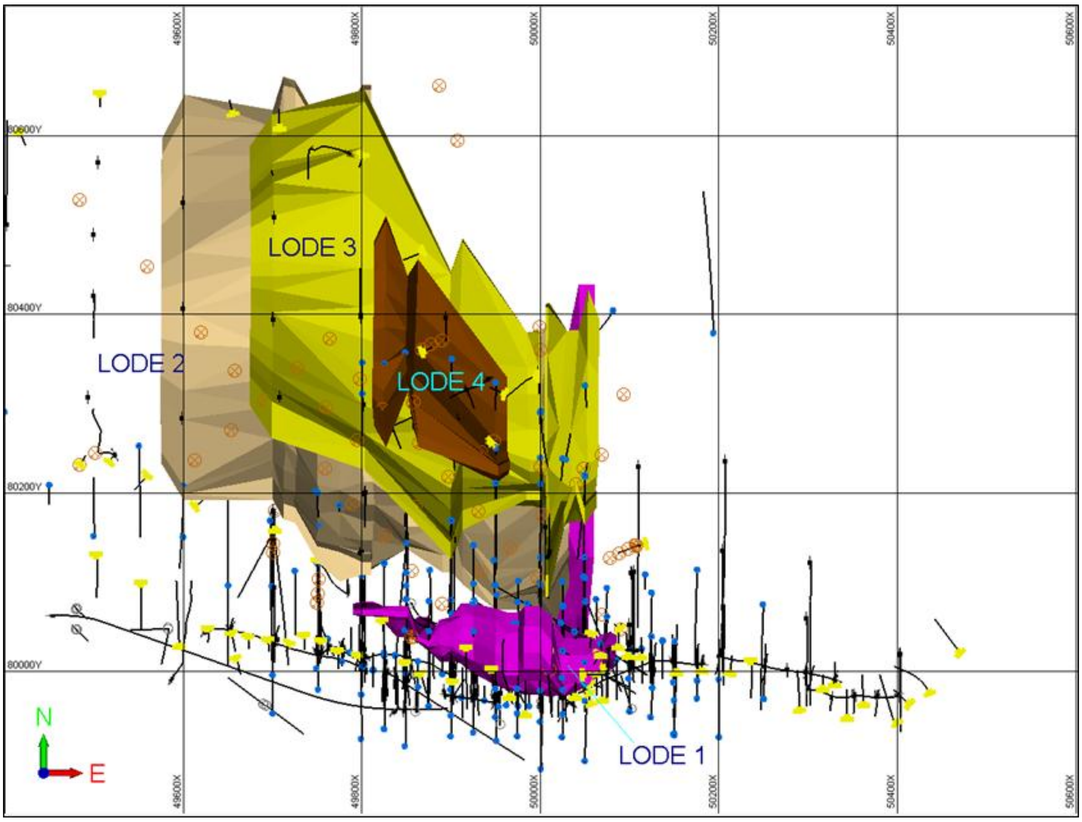


Figure 18. Plan view on extents of the modelled mineralized DISS lodes with drill holes collars

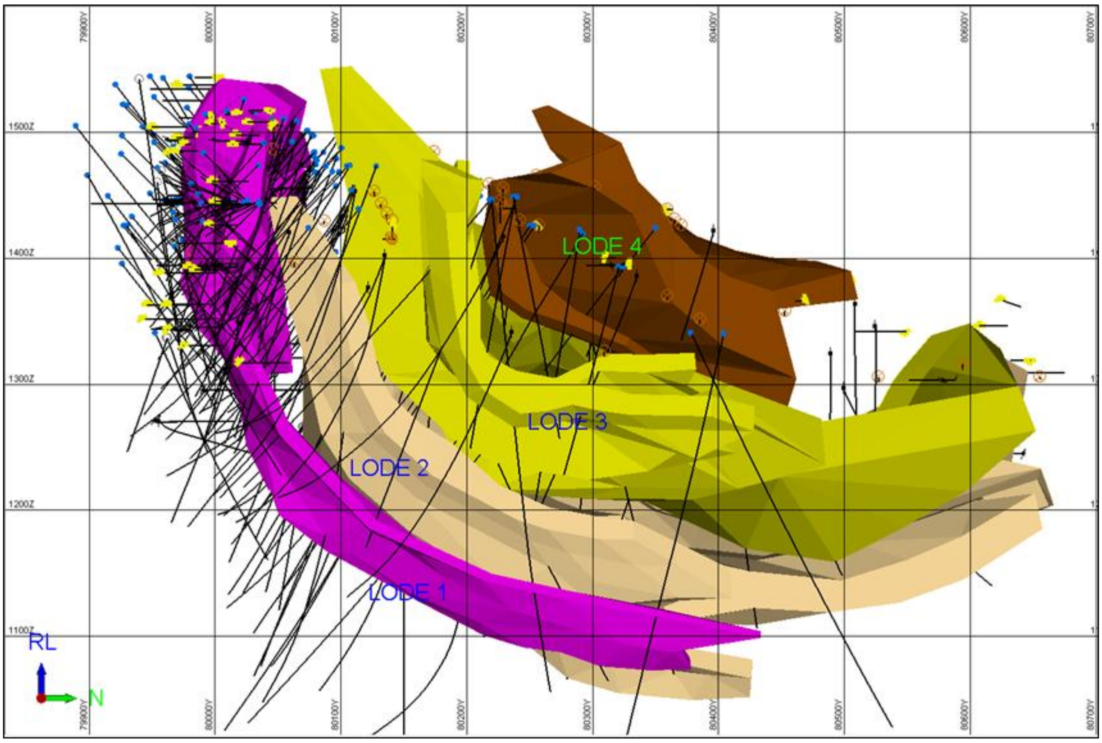


Figure 19. 2D view looking west on extents of DISS mineralized lodes with drill hole collars.

14.3 Sample Length Analysis and Compositing

14.3.1 Data Preparation

General aspects of data preparation used in resource estimation are as follows:

- Examination of raw sample lengths and selection of composite length.
- Compositing data to 1 m down-hole lengths, breaking the compositing at geological boundaries.

14.3.2 Selection of Composite Length

Analysis of the exploration data intervals concludes the majority of the raw sample intervals are 1 m or 2 m in length, but there are a number of non-regular sample data. The raw samples range in length from 0.05m to 3.0m (Appendix A), with about 90% being 1m. The 1m length was considered appropriate for compositing to retain the original data variability. Use of this composite size minimised splitting of raw samples to smaller intervals.

A composite interval of 1m was chosen to maintain the differentiation of both the lodes and the high grade zones within the individual lodes.

Data was composited to 1m down hole constrained by lode boundaries within each lode. The univariate statistics of 1m composite data are summarized in the next section.

Compositing was completed to honour the geological boundaries of the mineralized lode by breaking the composites at the lode boundaries. This process tends to create sub 1m samples at lode contacts (approximately 1% of the composites have a length less than or equal to 0.3m). Only diamond drill hole and high quality underground channel data were used in the analysis and estimate.

For the exploration-based estimates this was addressed by adapting the Kriging system to account for the sample lengths, so that all the available composites could be used in the estimation process.

Assay length distribution is shown in Figure 20 and composite length distribution in Figure 21 and Figure 22.

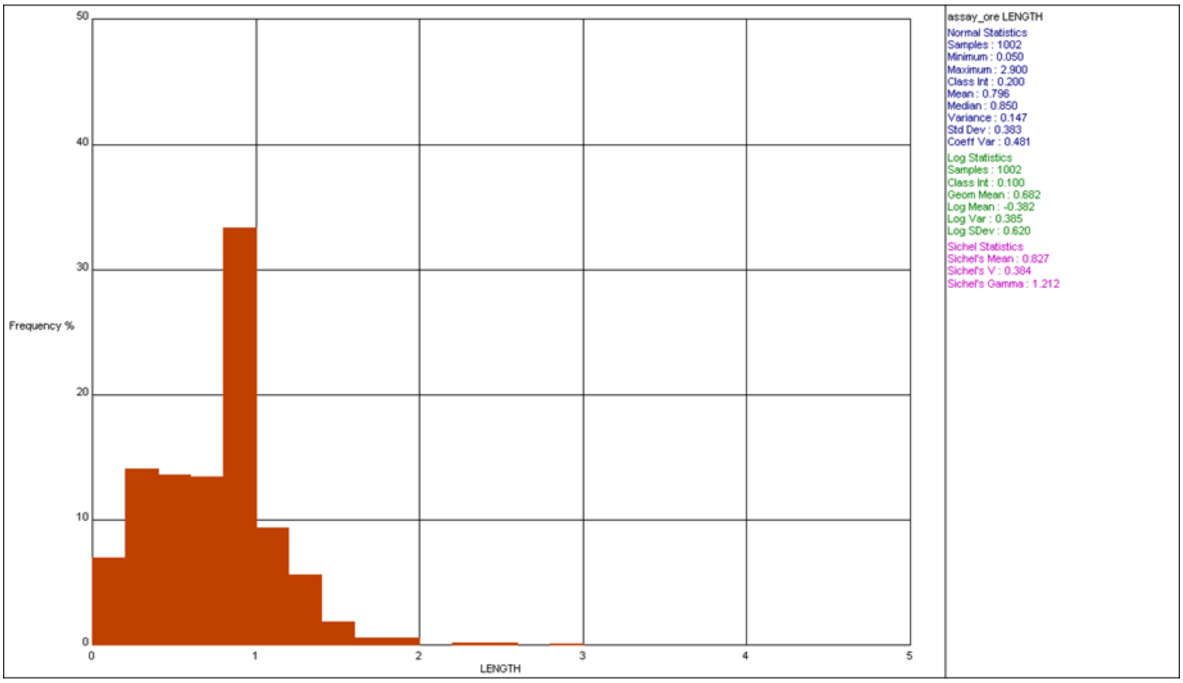


Figure 20. Assay length distribution

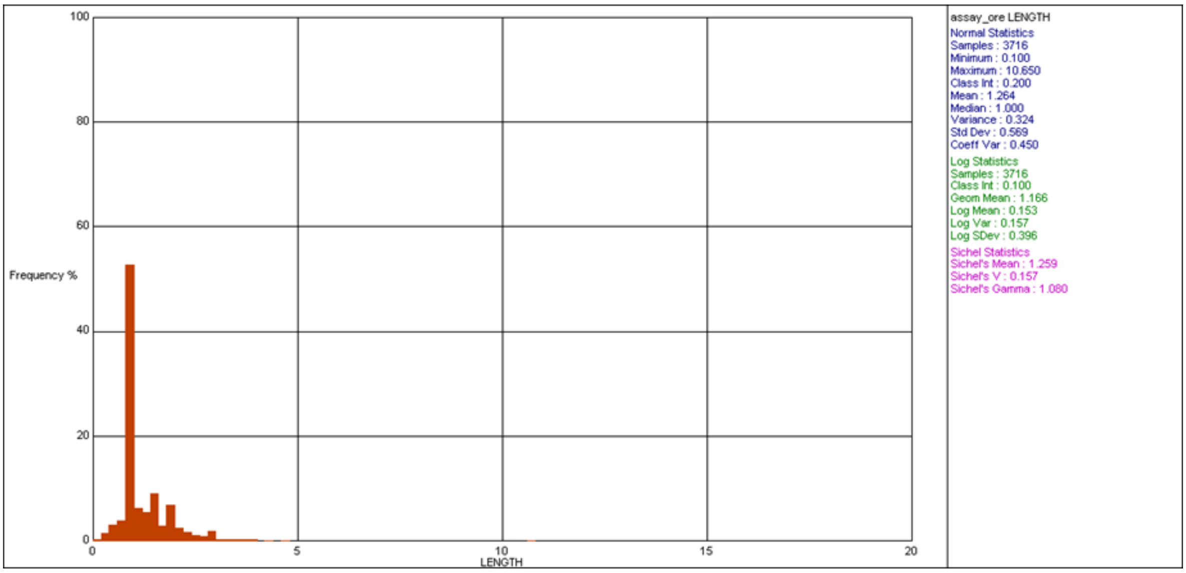
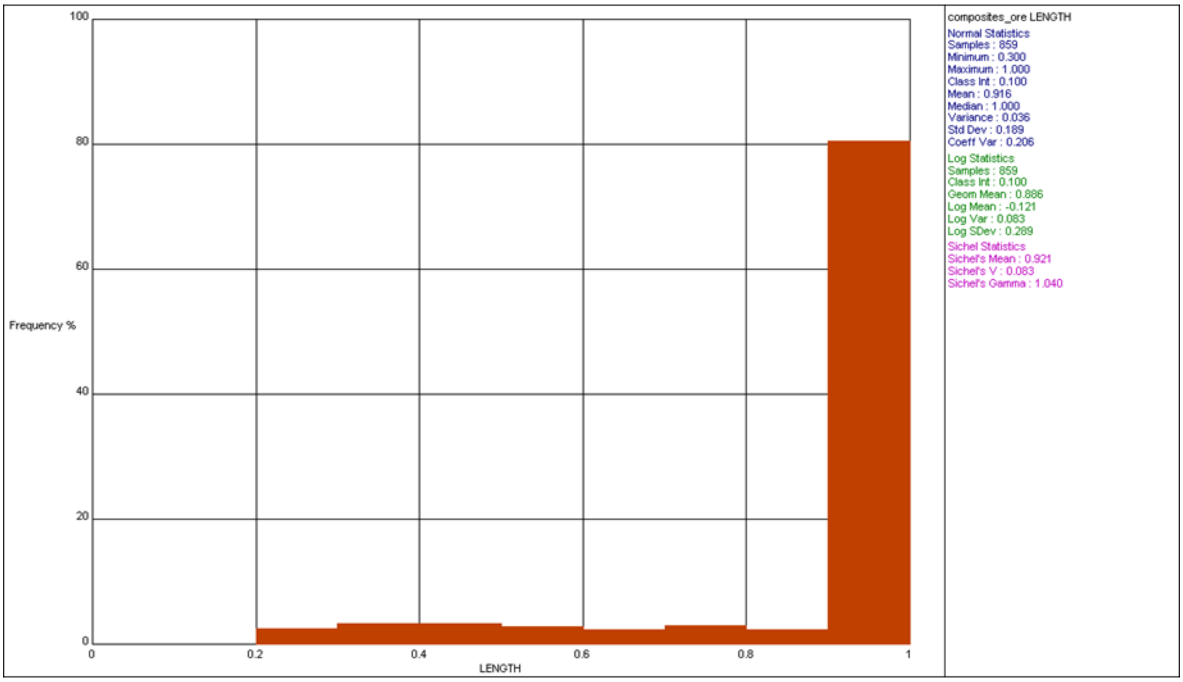


Figure 21. Ban Phuc MSV (upper) and DISS (lower) assay interval length histogram



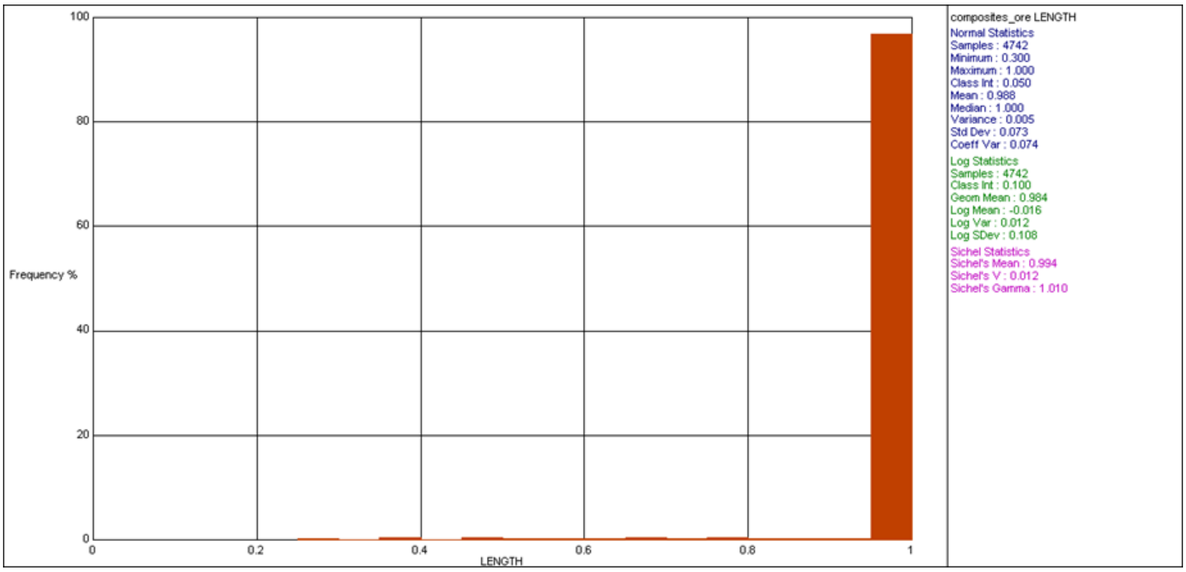


Figure 22. Ban Phuc MSV (upper) and DISS (lower) composite interval length histogram.

14.4 Statistical Analyses

14.4.1 Summary Statistics

The statistical analysis examined the distributions of the composited Ni, Cu, Co, S, Fe and Mg grades within each modelling lode, particularly the upper tail of the distributions. The univariate statistics and probability plots were generated for Ni, Cu, Co, S, Fe and Mg for each mineralization lode. The statistical summary for 1m composite Ni, Cu, Co, S, Fe and Mg uncut values are provided in Table 39 and Table 40.

The probability plots by lode are shown in Figure 23 and Figure 24 separately.

Table 39. Univariate statistics for 1m composite by MSV lode

Variable	Lode	Number	Minimum	Maximum	Mean	Median	Std Dev	Variance	Coeff Var
Ni	1	741	0.127	8.83	2.546	1.571	2.325	5.404	0.913
	2	118	0.15	8.716	3.371	2.87	2.574	6.627	0.764
Cu	1	741	0.013	35.9	1.123	0.84	1.599	2.555	1.423
	2	118	0.047	6.165	1.34	1.022	1.159	1.342	0.864
Co	1	579	0.003	0.263	0.076	0.047	0.068	0.005	0.894
	2	88	0.007	0.294	0.114	0.115	0.077	0.006	0.674
S	1	576	0.018	33.699	11.304	8.37	9.321	86.887	0.825
	2	88	0.83	34.663	14.899	14.234	10.046	100.912	0.674
MgO	1	481	0	43.918	7.465	2.24	8.909	79.377	1.193
	2	86	0.027	24.707	6.834	2.603	7.688	59.1	1.125
Fe	1	481	0	53.272	23.008	16.355	15.464	239.123	0.672
	2	86	6.634	54.6	31.966	34.63	16.271	264.737	0.509
SG	1	547	2.267	4.85	3.459	3.22	0.653	0.426	0.189
	2	88	2.661	4.688	3.877	3.93	0.653	0.426	0.168

Table 40. Univariate statistics for 1m composite by DISS lode

Variable	Lode	Number	Minimum	Maximum	Mean	Median	Std Dev	Variance	Coeff Var
Ni	1	2226	0.01	2.95	0.662	0.579	0.379	0.144	0.573
	2	1231	0.011	2.613	0.548	0.435	0.348	0.121	0.635
	3	1250	0.111	3.005	0.558	0.417	0.383	0.147	0.686
	4	112	0.244	2.539	0.869	0.613	0.579	0.335	0.666
Cu	1	1966	0	2.367	0.14	0.096	0.162	0.026	1.158
	2	372	0	0.792	0.068	0.027	0.107	0.011	1.563
	3	448	0	0.985	0.093	0.05	0.114	0.013	1.221
	4	107	0.001	0.365	0.108	0.075	0.097	0.009	0.905
Co	1	1895	0.001	0.068	0.016	0.014	0.007	0	0.444
	2	310	0.001	0.028	0.012	0.011	0.004	0	0.348
	3	378	0.004	0.069	0.014	0.012	0.007	0	0.505
	4	107	0.004	0.039	0.016	0.014	0.006	0	0.417
S	1	1849	0.001	16.512	0.973	0.51	1.272	1.617	1.307
	2	307	0.004	2.345	0.511	0.364	0.48	0.231	0.939
	3	372	0.001	10.101	0.685	0.364	0.84	0.706	1.226
	4	107	0.001	1.849	0.213	0.098	0.308	0.095	1.446
MgO	1	1304	0	45.793	32.259	34.988	9.596	92.081	0.297
	2	293	15.894	45.458	36.549	37.751	4.911	24.122	0.134
	3	289	0.697	45.41	36.948	37.222	4.033	16.267	0.109
	4	107	22.931	41.412	34.778	35.635	4.194	17.587	0.121

Variable	Lode	Number	Minimum	Maximum	Mean	Median	Std Dev	Variance	Coeff Var
Fe	1	1304	0	31.34	6.639	6.53	2.385	5.689	0.359
	2	293	3.66	8.74	5.628	5.49	1.031	1.063	0.183
	3	289	1.84	39.1	6.071	5.649	2.423	5.869	0.399
	4	107	4.007	14.57	6.51	5.865	2.269	5.15	0.349
SG	1	1807	1.07	3.465	2.622	2.619	0.152	0.023	0.058
	2	309	2.086	3.491	2.666	2.624	0.159	0.025	0.06
	3	369	2.027	3.382	2.65	2.635	0.169	0.028	0.064
	4	87	2.129	2.757	2.527	2.56	0.109	0.012	0.043

Statistical analyses of the 1m composites show Ni and other variables generally have coefficient variance (CV) below 1; Ni and Co, Ni and Fe, Ni and S, Ni and SG have a high correlation coefficient (Figure 23 and Figure 24).

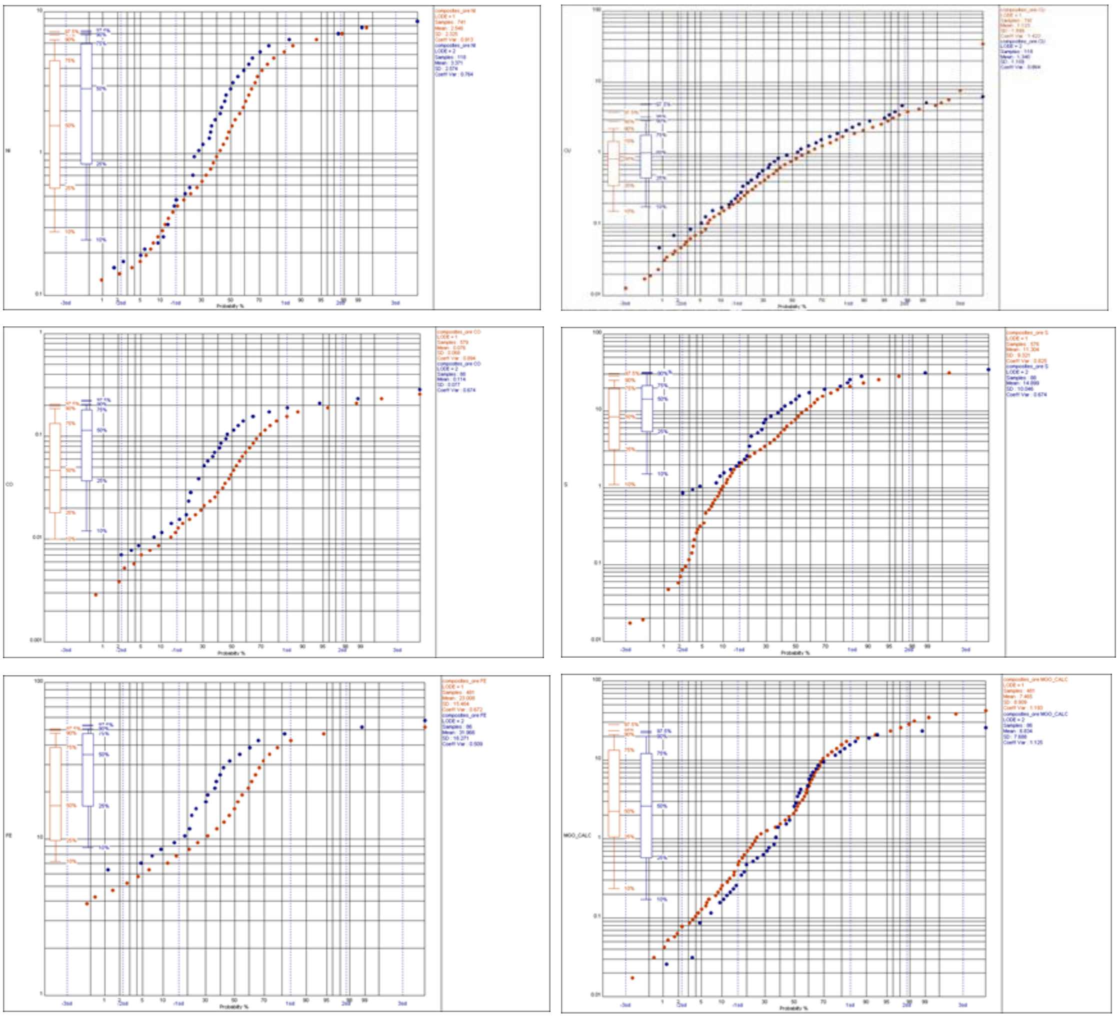


Figure 23. 1m composite of Ni, Co, Co, S, Fe and Mg probability plots by lode of MSV (Lode1-brown; Lode2-blue)

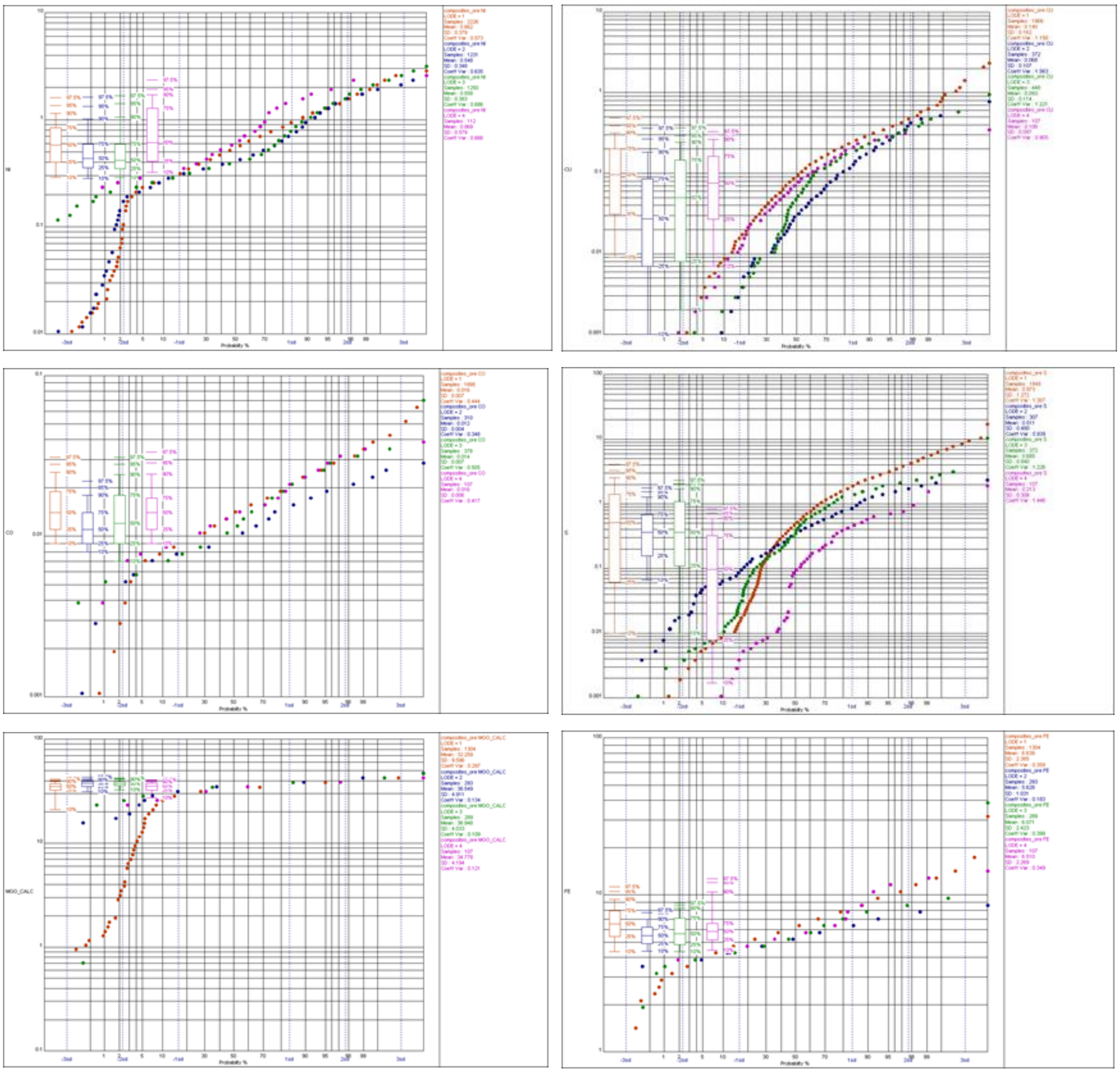


Figure 24. 1m composite of Ni, Co, Co, S, Fe and Mg probability plots by lode of DISS (Lode1-brown; Lode2-blue; Lode3-green; Lode4-purple)

14.4.2 Treatment of Outliers

All linear, non-linear interpolation methods, such as Ordinary Kriging (OK) and Multiple Indicator Kriging (MIK) are sensitive to the presence of high-grade outliers. For the resource estimation, the high-grade outliers were treated using a spatial constraining approach, which required a spatial constraining of high grades to individual blocks to control extrapolation of high-grade.

Selections of the outlier thresholds was carried out through visual inspection of log-normal cumulative probability plots above to determine naturally occurring breaks in the upper tail of the exploration sample populations. The high-grade thresholds for the estimates are shown in Table 41.

Table 41. High-Grade Thresholds

Lode	MSV High-Grade Threshold					
	Ni (%)	Cu (%)	Co (%)	S (%)	Fe (%)	Mg (%)
1	6.5	6	na	25	45	20
2	6.5	6	na	25	45	20
Lode	DISS High-Grade Threshold					
	Ni (%)	Cu (%)	Co (%)	S (%)	Fe (%)	Mg (%)
1	2.0	2.0	na	10	35	15
2	2.0	2.0	na	10	35	15
3	2.0	2.0	na	10	35	15
4	2.0	2.0	na	10	35	15

14.5 Flattening

14.5.1 Objectives

A ‘flattening’ or an ‘unfolding’ process has been carried out prior to variography and interpolation. The objectives are aimed at removing any variable dip and strike typically associated with the mineralized lodes.

14.5.2 Flattening Procedure

The “Flattening” technique for the composite dataset was applied using Micromine software. The data was flattened relative to the centre line of the lode wireframe. The process of flattening allows a variable to have constant plunge and dip directions and essentially reduces the variogram search direction to two dimensions. The major and semi-major axes are now defined as the direction of greatest continuity in the flattened plane.

The effect of flattening for composites and model as examples for MSV is shown in Figure 25.

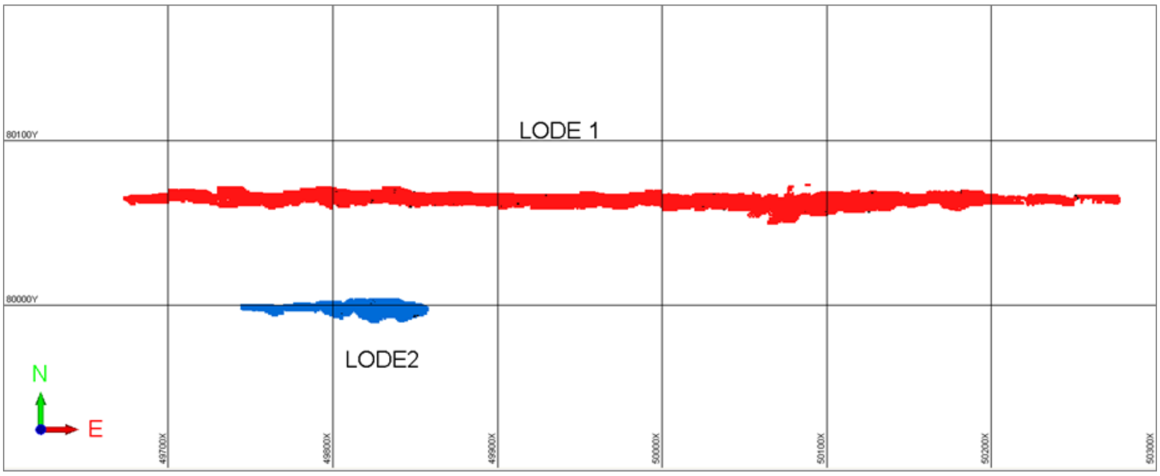


Figure 25. A plan view for the Flatten Model and Composite Data (block) for MSV

14.6 Variography

14.6.1 Objectives

The objectives of the variography analysis are to:

- establish the directions of major grade continuity for Ni, Cu, Co, S, Fe, Mg and SG in the ore domains; and
- provide variogram model parameters for use in geostatistical grade interpolation

Because of the sparse data in Lode 2, the variography was based on the 1m composite of exploration data for the main lode (Lode 1).

14.6.2 Variography Analysis Procedure

Variography and evaluation of suitable estimation parameters based on the final variogram models were undertaken using GeoAccess software. The variography analysis was based on the "flattened" data of the major lode, and the variogram model parameters have been used to represent the minor lode.

Variography has been carried out using a three-dimensional directional approach. The angular and distance search tolerances used are provided in Table 42, which illustrate the various tolerances for directional variogram calculation.

Down hole variograms are used to determine the nugget effect, then a fan of horizontal variograms is used to select major and semi-major variograms; these will usually be aligned with (major) and at right angles (semi-major) to the strike of the mineralized domains. A vertical or down hole variogram can then be used for the down-dip direction.

Table 42. The Parameters used for variogram generation

Parameter	Value
Start Azimuth (ang)	0°
End Azimuth	175°
Step Azimuth	5°
Horizontal Angle Tolerance (atol)	15°
Horizontal Angular Increment	5°
Lag Distance (xlag)	10m
Lag Tolerance (xltol)	5m

***NB**: Azimuth (horizontal direction vector) is defined as a clockwise bearing from grid north (0°). Plunge (vertical direction vector) is defined from the horizontal plane (0°), where a negative plunge is down and positive is up.*

An overview of the variography procedure for 1 m composite dataset is as follows:

- Subset oxide and fresh 1 m composite with top cut dataset were applied for the variography analysis;

- The nugget variances were modelled from the down hole variograms based on a 1m lag interval;
- Variograms were generated with a 10 m lag interval and used to select major and semi-major variograms, with 5° increments horizontally to provide complete vector coverage in 2D;
- Normal variograms were used as these generally produced the clearest variogram structure compared to absolute variograms and pair-wise relative variograms;
- Variogram map trends were related back to the expected geological continuity directions. Orientations were modelled consistent with the interpretations and supported by the continuity trends indicated by the variography and geological understanding;
- The two orthogonal orientations which best reflected the major and semi major axes of continuity were selected; A vertical or down hole variogram can then be used for the minor direction.

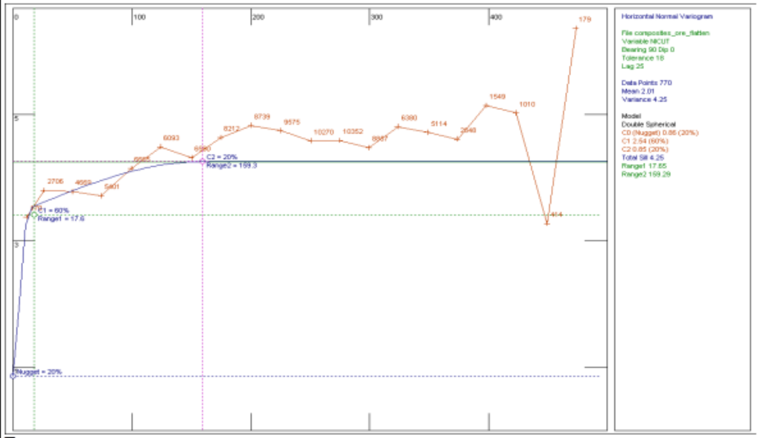
The variograms were calculated for 'Ni_Cut', 'Cu_Cut', 'Co_Cut', 'S_Cut', 'Fe_Cut', 'Mg_Cut' and SG using the above approach. Variogram modelling was then carried out; in general, a double spherical scheme model was adequate to represent the raw variograms and defined the shorter and longer scale variability. As a final step, the variogram models were normalised to a variance of 1 to facilitate comparison of Kriging variances after interpolation.

Nugget effect in the major lodes is typically 20% to 25%, which is moderate for a nickel and copper deposit and illustrates the robustness of the unfolded coordinate system as used for variogram calculation. Major ranges varied from 120m to 150m, with a limited range across the mineralization of typically 15 to 35m. Down plunge ranges can be limited to 6m to 15m in some cases. The MSV Lode1 has data spacing that is well within the variogram ranges.

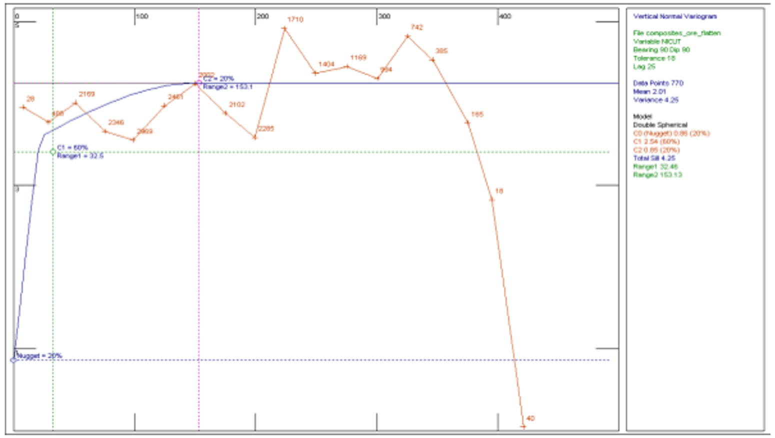
The Ni variogram plots for MSV Lode1 have been shown in Figure 26 as an example. The final variogram parameters selected for the Ordinary Kriging process are summarized in Table 43.

Ni Variogram Parameters MSV_LODE1

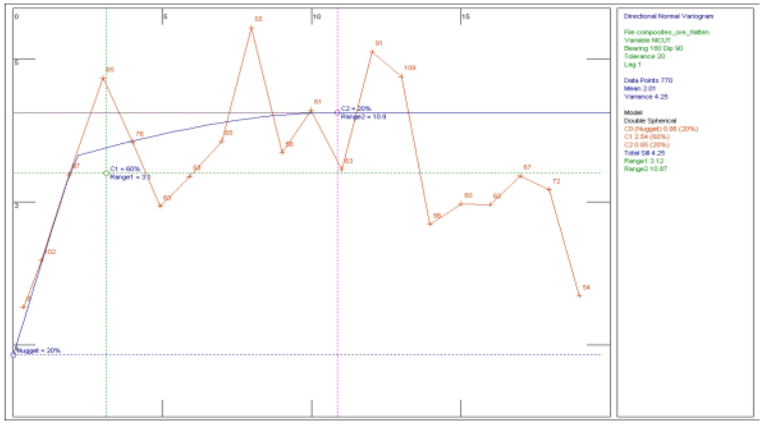
X



Y



Z



Normalised Variogram Parameters

Nugget	0.20
C1	0.60
C2	0.20
Sill	1.00

Ranges	X	18	159
Ranges	Y	32	153
Ranges	Z	3	11

Figure 26. Experimental Variograms and Models along-strike, across-strike and down-dip directions for MSV Ni Lode1

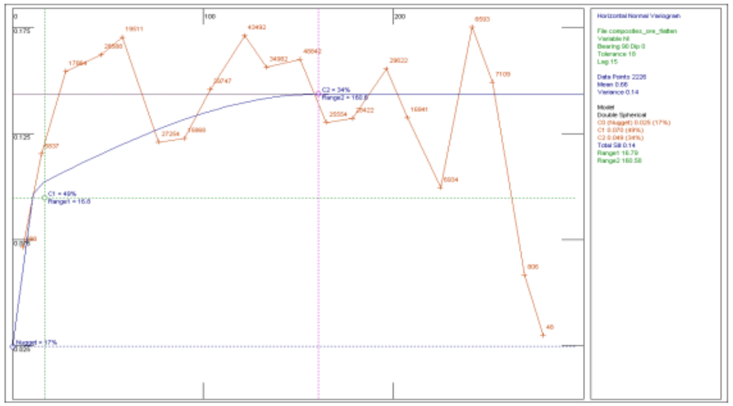
Table 43. Variogram Model Parameters for MSV.

Variable	Lode	Direction	Nugget	C1	C2	Sill	Range1	Range2	
Ni	1	Across-Strike	X	0.22	0.58	0.20	1.00	19.47	156.89
		Along-strike	Y	0.22	0.58	0.20	1.00	22.43	140.27
		Down-dip	Z	0.22	0.58	0.20	1.00	2.83	8.64
Cu	1	Across-Strike	X	0.23	0.56	0.21	1.00	31.23	190.07
		Along-strike	Y	0.23	0.56	0.21	1.00	27.42	119.19
		Down-dip	Z	0.23	0.56	0.21	1.00	2.62	5.78
Co	1	Across-Strike	X	0.23	0.58	0.19	1.00	23.95	155.90
		Along-strike	Y	0.23	0.58	0.19	1.00	44.88	120.83
		Down-dip	Z	0.23	0.58	0.19	1.00	3.33	6.40
S	1	Across-Strike	X	0.25	0.52	0.23	1.00	37.20	172.49
		Along-strike	Y	0.25	0.52	0.23	1.00	52.84	127.68
		Down-dip	Z	0.25	0.52	0.23	1.00	4.13	6.87
Fe	1	Across-Strike	X	0.12	0.48	0.40	1.00	12.91	120.37
		Along-strike	Y	0.12	0.48	0.40	1.00	15.38	93.65
		Down-dip	Z	0.12	0.48	0.40	1.00	8.99	17.59
Mg	1	Across-Strike	X	0.11	0.60	0.29	1.00	24.03	105.19
		Along-strike	Y	0.11	0.60	0.29	1.00	12.35	89.43
		Down-dip	Z	0.11	0.60	0.29	1.00	9.35	13.71
SG	1	Across-Strike	X	0.15	0.42	0.43	1.00	18.54	156.42
		Along-strike	Y	0.15	0.42	0.43	1.00	12.01	95.91
		Down-dip	Z	0.15	0.42	0.43	1.00	2.94	3.89

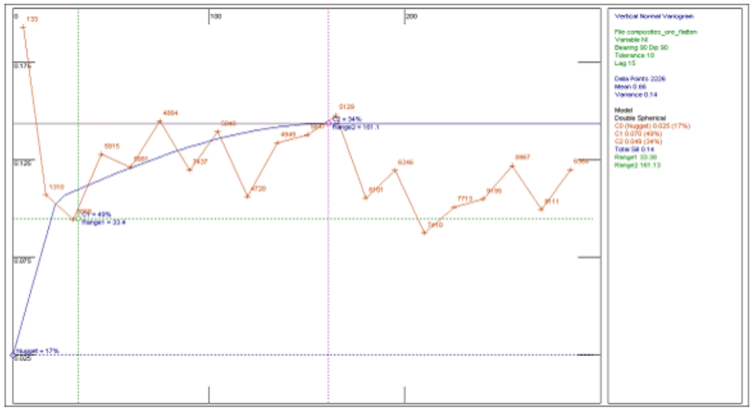
The Ni variogram plots for DISS Lode1 – 3 have been shown in Figure 27 as an example. The final variogram parameters selected for the Ordinary Kriging process are summarized in Table 44.

Ni Variogram Parameters DISS_LODE1

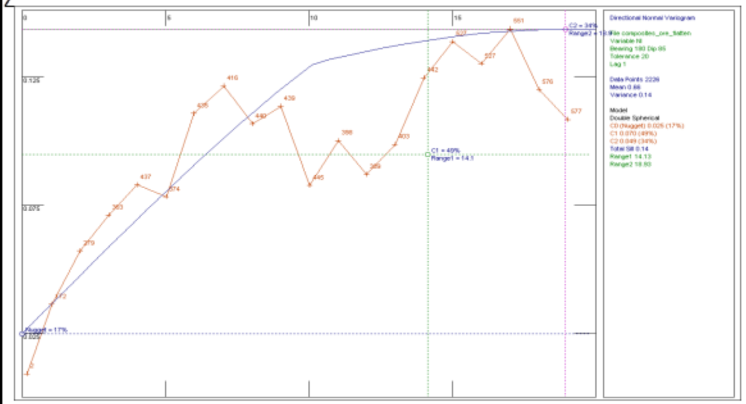
X



Y



Z



Normalised Variogram Parameters

Nugget	0.17
C1	0.49
C2	0.34
Sill	1.00

Ranges	X	17	161
Ranges	Y	33	161
Ranges	Z	14	19

Figure 27. Experimental Variograms and Models along-strike, across-strike and down-dip directions for DISS Ni Lode1

Table 44. Variogram Model Parameters for DISS Domain

Variable	Lode	Direction	Nugget	C1	C2	Sill	Range1	Range2	
Ni	1	Across-Strike	X	0.17	0.49	0.34	1.00	16.79	160.58
		Along-strike	Y	0.17	0.49	0.34	1.00	33.38	161.13
		Down-dip	Z	0.17	0.49	0.34	1.00	14.13	18.93
Cu	1	Across-Strike	X	0.25	0.46	0.29	1.00	8.82	125.66
		Along-strike	Y	0.25	0.46	0.29	1.00	16.82	111.92
		Down-dip	Z	0.25	0.46	0.29	1.00	1.13	6.41
Co	1	Across-Strike	X	0.24	0.36	0.40	1.00	8.13	137.74
		Along-strike	Y	0.24	0.36	0.40	1.00	24.92	186.08
		Down-dip	Z	0.24	0.36	0.40	1.00	16.15	29.97
S	1	Across-Strike	X	0.11	0.73	0.17	1.00	15.64	46.54
		Along-strike	Y	0.11	0.73	0.17	1.00	46.23	271.20
		Down-dip	Z	0.11	0.73	0.17	1.00	21.94	22.56
Fe	1	Across-Strike	X	0.02	0.59	0.39	1.00	22.59	86.96
		Along-strike	Y	0.02	0.59	0.39	1.00	19.87	75.33
		Down-dip	Z	0.02	0.59	0.39	1.00	27.94	57.73
Mg	1	Across-Strike	X	0.02	0.62	0.36	1.00	21.32	71.56
		Along-strike	Y	0.02	0.62	0.36	1.00	17.83	114.79
		Down-dip	Z	0.02	0.62	0.36	1.00	34.32	48.32
SG	1	Across-Strike	X	0.25	0.51	0.24	1.00	23.64	158.47
		Along-strike	Y	0.25	0.51	0.24	1.00	38.54	133.90
		Down-dip	Z	0.25	0.51	0.24	1.00	1.61	4.42
Ni	2	Across-Strike	X	0.20	0.30	0.50	1.00	52.79	187.64
		Along-strike	Y	0.20	0.30	0.50	1.00	11.37	127.68
		Down-dip	Z	0.20	0.30	0.50	1.00	8.59	25.44
Cu	2	Across-Strike	X	0.22	0.50	0.28	1.00	68.80	174.73
		Along-strike	Y	0.22	0.50	0.28	1.00	68.80	136.12
		Down-dip	Z	0.22	0.50	0.28	1.00	25.56	29.87
Co	2	Across-Strike	X	0.20	0.49	0.31	1.00	73.00	178.66
		Along-strike	Y	0.20	0.49	0.31	1.00	96.96	128.16
		Down-dip	Z	0.20	0.49	0.31	1.00	144.80	164.30
S	2	Across-Strike	X	0.11	0.18	0.71	1.00	40.38	133.85
		Along-strike	Y	0.11	0.18	0.71	1.00	53.05	131.43
		Down-dip	Z	0.11	0.18	0.71	1.00	5.23	36.42
Fe	2	Across-Strike	X	0.11	0.32	0.57	1.00	80.15	260.41
		Along-strike	Y	0.11	0.32	0.57	1.00	80.15	167.41
		Down-dip	Z	0.11	0.32	0.57	1.00	148.53	254.80
Mg	2	Across-Strike	X	0.06	0.49	0.45	1.00	120.46	189.78
		Along-strike	Y	0.06	0.49	0.45	1.00	68.31	131.44
		Down-dip	Z	0.06	0.49	0.45	1.00	151.32	169.49

Variable	Lode	Direction	Nugget	C1	C2	Sill	Range1	Range2	
SG	2	Across-Strike	X	0.29	0.19	0.52	1.00	37.80	142.42
		Along-strike	Y	0.29	0.19	0.52	1.00	37.80	78.43
		Down-dip	Z	0.29	0.19	0.52	1.00	28.11	62.39
Ni	3	Across-Strike	X	0.09	0.38	0.53	1.00	26.52	155.27
		Along-strike	Y	0.09	0.38	0.53	1.00	36.81	74.08
		Down-dip	Z	0.09	0.38	0.53	1.00	36.51	39.43
Cu	3	Across-Strike	X	0.10	0.18	0.72	1.00	40.21	98.89
		Along-strike	Y	0.10	0.18	0.72	1.00	40.21	92.62
		Down-dip	Z	0.10	0.18	0.72	1.00	5.52	52.58
Co	3	Across-Strike	X	0.05	0.26	0.69	1.00	44.50	122.78
		Along-strike	Y	0.05	0.26	0.69	1.00	11.86	79.15
		Down-dip	Z	0.05	0.26	0.69	1.00	66.64	91.06
S	3	Across-Strike	X	0.15	0.22	0.63	1.00	39.50	139.64
		Along-strike	Y	0.15	0.22	0.63	1.00	6.16	44.39
		Down-dip	Z	0.15	0.22	0.63	1.00	5.81	39.10
Fe	3	Across-Strike	X	0.05	0.42	0.53	1.00	80.43	173.52
		Along-strike	Y	0.05	0.42	0.53	1.00	38.11	100.31
		Down-dip	Z	0.05	0.42	0.53	1.00	123.99	178.52
Mg	3	Across-Strike	X	0.10	0.23	0.67	1.00	58.50	148.30
		Along-strike	Y	0.10	0.23	0.67	1.00	52.97	123.11
		Down-dip	Z	0.10	0.23	0.67	1.00	126.48	149.27
SG	3	Across-Strike	X	0.06	0.49	0.45	1.00	19.83	126.83
		Along-strike	Y	0.06	0.49	0.45	1.00	61.64	115.18
		Down-dip	Z	0.06	0.49	0.45	1.00	31.93	54.98

14.7 Block Model

14.7.1 Block Model Extents and Block Size

Block Model size and location was determined to ensure complete coverage of any likely area of interest for optimisation work. The model extents are summarized in Table 45.

Table 45. Block Model Geometry Summary

Project	Direction	Minimum	Maximum	Block Size	Sub Blocking	Rotation
Ban Phuc MSV	Easting	49600	50400	10	1	0
	Northing	79900	80150	1	0.2	0
	RL	1000	1600	5	1	0
Ban Phuc DISS	Easting	49500	50200	10	1	0
	Northing	79900	80700	5	1	0
	RL	1000	1800	5	1	0

Drillhole data spacing, mining selectivity and mineralized lode geometry are among the primary considerations for the determination of an appropriate estimation block size. Drilling data at Ban Phuc is primarily on a 25 × 25 and 25 × 50 metre drilling patterns, grading to a 30 × 30 to 30 × 50 metre patterns at depth.

Sub-cells were generated as appropriate to honour wireframe domains and regolith interpretations during model construction.

14.7.2 Block Model Creation

The empty block model was constructed using Micromine software. The model was initially created as separate geological block models with varying sub-block resolution for ore, waste, dump, weathering and mining boundaries whilst maintaining a majority (parent cell) assigning approach for the Ban Phuc MSV deposit.

Blocks were generated to parent cell sizes and sub-celled where necessary based on interpretation wireframes. Attributes and zones were built into the model as summarized below in Table 46.

Table 46. Summary of attributes for Ban Phuc MSV and DISS Block Model

Attribute	Code	Decimals	Description
Domain	Text	0	Individual domain (lithology) code for estimation
Lode	Numeric	2	Individual lode code for estimation
Topo	Text	0	Topographical code: Air; Mined; Fill; or InSitu. **
Regolith	Text	0	Regolith profile assigned by Wireframe: Air; Fill; Oxide; Transitional; Fresh
Material	Text	0	Material code: Air; Fill; Oxide; Transitional; Fresh
Rescat	Numeric	0	Resource classification: 1- Measured; 2 – Indicated; 3 – Inferred; 4 - unclassified
Ni	Numeric	3	Ordinary kriged estimate using cut assay values
Cu	Numeric	3	Ordinary kriged estimate using uncut assay values
Co	Numeric	3	Ordinary kriged estimate using cut assay values
S	Numeric	3	Ordinary kriged estimate using uncut assay values
Fe	Numeric	3	Ordinary kriged estimate using cut assay values
Mg	Numeric	3	Ordinary kriged estimate using uncut assay values
SG	Numeric	3	Ordinary kriged estimate using uncut assay values

***A separate model with the mined codes replaced with air and densities of zero was created for the mine planning department.*

14.7.3 Kriging Neighbourhood Analysis

Quantitative Kriging Neighbourhood Analysis (QKNA) was undertaken on a subset of blocks in the main domains to establish optimum search and minimum/maximum composite parameters. Goodness-of-fit statistics are generated to assess the efficiency of the various parameters. The primary statistics used are the Kriging efficiency and the slope of regression.

A general summary of the main steps is provided:

- Run QKNA for a range of potential Kriging neighbourhoods;
- Produce summary graphs for QKNA criteria (Kriging slope of regression, sum of negative Kriging weights, Kriging efficiency);
- Select Kriging neighbourhoods as per QKNA optimisation theory.

***Note** : The Kriging efficiency is calculated as (block variance-Kriging variance)/block variance, where block variance is the total sill less the variance contained within a block.*

The slope of regression is calculated as (block variance – Kriging variance + μ)/ (block variance – Kriging variance + 2μ).

Kriging efficiency (KE) calculates the overlap expected between the estimated block grade histogram and the 'true' block grade histogram. A high efficiency indicates a good match between estimated and 'true' grades, while as parameters become less optimal, KE drops.

The slope of regression estimates the correlation between estimated and 'true' grades; a value closer to 1.0 indicates a good fit. In addition, other statistics, such as the percentage of negative weights generated in a Kriging plan can be considered. A number of key input parameters can be tested in this way, including:

- Block size.
- Number of discretisation points.
- Search ellipse dimensions.

Minimum and maximum sample numbers in a search plan.

A summary of the selected Kriging neighbourhoods by lode used in the Ordinary Kriging is provided in Table 47 and Table 48. Also summarized is the maximum sample limit applied for OK estimation of each domain with the selected neighbourhoods. The Kriging search parameters are applied in the estimation of each domain.

Minimum number of samples, numbers of drill holes, and search distances are determined by drill pattern spacing, and the geometry of the mineralized lodes. Ban Phuc MSV mineralization occurs in relatively thin tabular lodes, often 3 - 6 meters in width, so a minimum of 8 samples per drill hole, in 4 drill holes was selected for the first search pass. The subsequent passes are set to lower minimums while increasing the search distances to find sufficient samples where drilling density decreases.

14.8 Grade Estimation

Grade estimation of the Ban Phuc deposit was carried out using the geostatistical method of Ordinary Kriging (OK). This method uses estimation parameters defined by the variography. The 1m composite with High Grade Constraining dataset was used for the grade interpolation. Estimation of the resource was completed using Datamine v3.17 software.

The Micromine format rock and grade models and composite data files were exported to Datamine software to use the ESTIMA interpolation process. Datamine software is a robust and flexible resource estimation software, providing maximum control over all the parameters which are used to drive the estimation process. Results are also reproducible and auditable through extensive use of macros and parameter files to control the process.

The Kriging plan used for resource estimation used a spatial restraining of high-grade (outlier) composites. This included a preliminary flagging pass to identify individual blocks containing the designated high-grade outlier values (based on the thresholds detailed in Table 47).

The Kriging plan parameters used for grade interpolation are summarized in Table 47 and Table 48. Specific search ellipsoid rotations were used for each domain reflecting the domain variography orientations. A 3-pass kriging plan was used to estimate blocks which did not receive a grade estimate in a previous pass. Search ellipsoid dimensions were selected in relation to the nominal drill hole data spacing and identified variogram ranges.

The geometry of the search ellipsoid was adjusted for MSV lodes to reflect the sub vertical mineralization and prevalence of drill holes down-structure, although similar search distances to supergene were used.

Table 47. Grade Interpolation Search Parameters for MSV

Variable	Lode	Search Ellipse			Search Pass 1		Search Pass 2			Search Pass 3		
		Major	Semi-Major	Minor	Min Samples	Max Samples	Search Factor	Min Samples	Max Samples	Search Factor	Min Samples	Max Samples
Ni	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
Cu	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
Co	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
S	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
Fe	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
Mg	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24
SG	1	60	30	10	8	24	2	8	24	3	2	24
	2	60	30	10	8	24	2	8	24	3	2	24

Table 48. Grade Interpolation Search Parameters for DISS

Variable	Lode	Search Ellipse			Search Pass 1		Search Pass 2			Search Pass 3		
		Major	Semi-Major	Minor	Min Samples	Max Samples	Search Factor	Min Samples	Max Samples	Search Factor	Min Samples	Max Samples
Ni	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24
Cu	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24
Co	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24
S	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24
Fe	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24

Variable	Lode	Search Ellipse			Search Pass 1		Search Pass 2			Search Pass 3		
		Major	Semi-	Minor	Min	Max	Search	Min	Max	Search	Min	Max
			Major		Samples	Samples	Factor	Samples	Samples	Factor	Samples	Samples
Mg	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24
SG	1	10	60	40	8	24	2	8	24	3	2	24
	2	40	60	20	8	24	2	8	24	3	2	24
	3	40	60	20	8	24	2	8	24	3	2	24
	4	40	60	20	8	24	2	8	24	3	2	24

The sequence of estimation passes for the method of spatial constraining high-grade are as detailed below.

1. A 5.1m by 0.6m by 2.6m nearest neighbour search radius was used to flag nominal 10m by 1m by 5m blocks (or smaller) near samples with Ni grades above designated high-grade thresholds. This approach essentially flagged blocks that contained the 'high grade' data, so that their influence could be spatially restricted to a volume equivalent to a nominal 10m by 1m by 5m block.

2. Grades were estimated for blocks not flagged in Step 1 using the Pass 1 nearest neighbour search. Only samples less than the Ni high-grade thresholds were used. This estimated blocks other than the 'high-grade flagged blocks' using data that excluded the 'high-grade' data.

3. Grades were estimated for blocks flagged in Step 1 using the Pass 1 nearest neighbour search (see Step 1). All the data was used including the 'high-grade' data. This estimated the 'high-grade flagged blocks' using not only the 'high-grade' data contained in those blocks, but also incorporating the influence of surrounding grade values.

4. Step 2 and 3 were repeated for un-estimated blocks using the Pass 2 and 3 search parameters. This was used to estimate blocks that were not estimated in the Pass 1 search (i.e. to fill gaps). Blocks estimated in this pass are flagged as pass2 blocks.

5. Block discretisation was set to 5 (X) by 2 (Y) by 5 (Z) to estimate block grades of 10m by 1m by 5m parent blocks.

The size of the sub-cells in the block model with respect to the general data spacings meant that estimation of sub-cell blocks was likely to entail a high degree of estimation error. Estimation of sub-cells in the model was performed to the parent cell size, so sub-cells also received the parent cell estimate.

14.9 Block Model Validation

Statistical and visual assessment of the block model was undertaken to assess successful application of the various estimation passes, to ensure that as far as the data allowed all blocks within domains were estimated and the model estimates considered acceptable.

14.9.1 Onscreen Validation

Each lode is checked against the composited data used in the estimation process. The onscreen validation process involved comparing block estimates and composites grades in cross section.

- The onscreen validation sections showed a strong correlation between the block and composite drill hole grade;
- There were no un-estimated blocks present within the ore lodes.

Example sections are shown below (See Figure 28 to Figure 30).

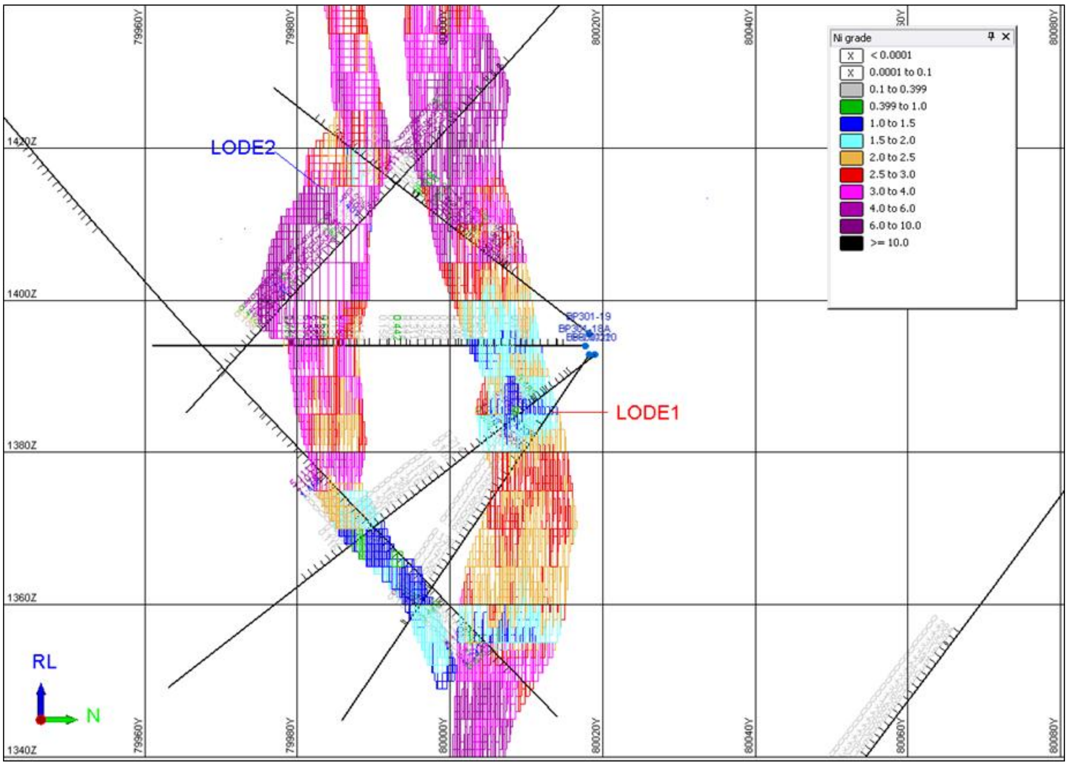


Figure 28. Cross section 49825E for Block Model Validation

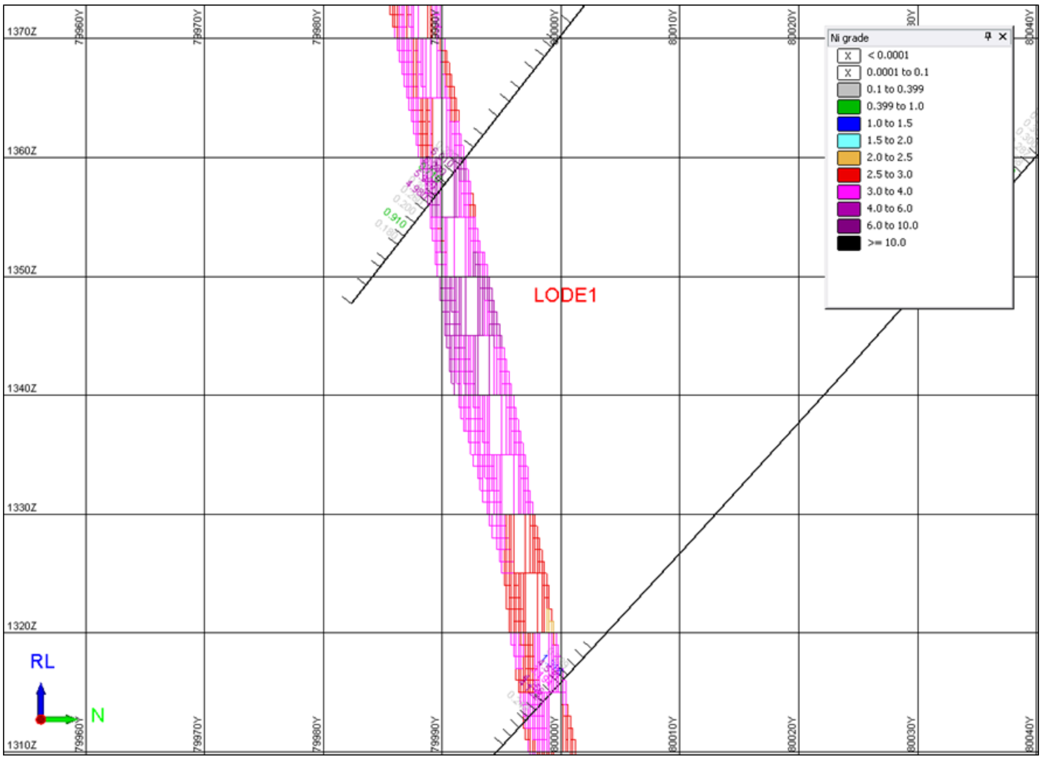


Figure 29. Cross section 49875E for Block Model Validation for MSV

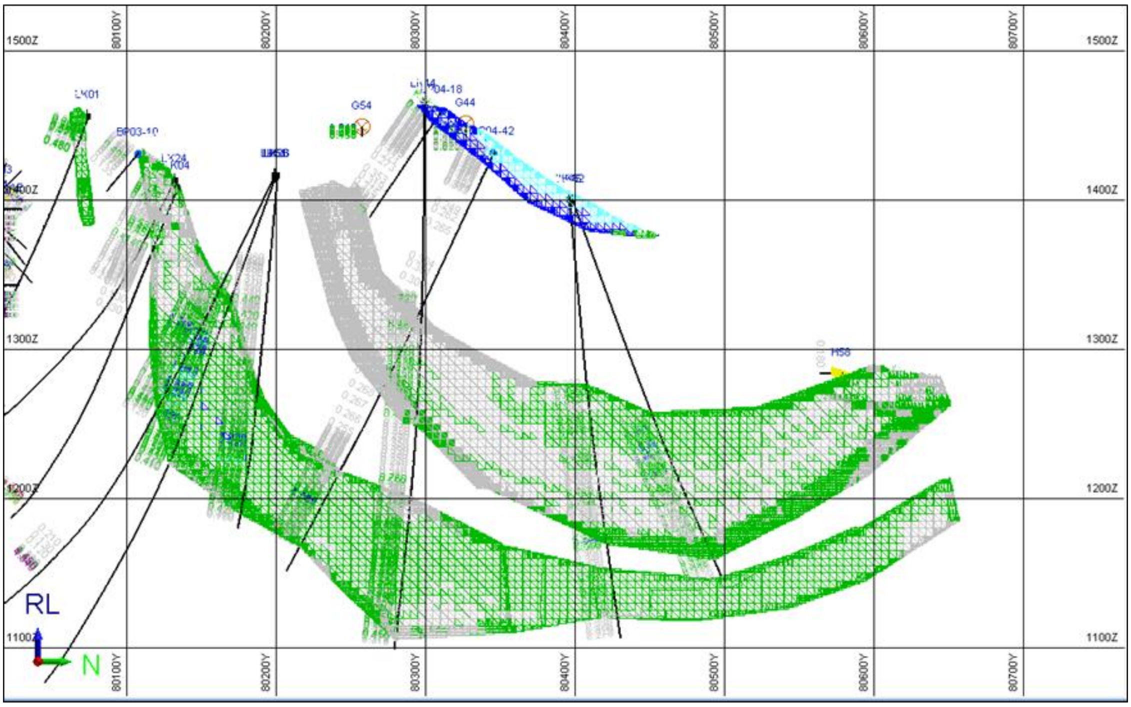


Figure 30. Cross section 49810E for Block Model Validation for DISS

14.9.2 Plots Validation of Interpolated Grades

The process involved averaging both the blocks and samples in panels of 20 m (easting) by 5 m (northing) by 10 m (RL). Comparisons were made along easting, northing and RL slices for the entire deposit are presented in Figure 31 to Figure 36.

The observations are summarized as:

- Generally good agreement is observed between the data and block model mean grade for all variables for easting, northing and RL slices. For average grade conformance, three domains in Low Grades and High Grades display comparable performance relative to the data;
- QQ and scatter plots for the averaged sample data vs. block model results show deviation from the 45° line, with overstatement of Low Grades and understatement of High Grades by the block model. This is a natural expected behaviour of moving from sample size data to a much bigger volume as represented by Kriged models. This effect is more pronounced for the Low Grade Ore for which smoothing is higher;
- The grades estimated for the individual blocks and composite assay dataset compared reasonably well globally and for all mineralized domains.

Overall, the plot validation process shows that the block model estimates follow the trend of the 1m composite grades across the deposit. Estimation smoothing is present to a larger degree in low grade areas, but is more controlled in the more constrained mineralized zones. The model is considered to be a satisfactory representation of the global Mineral Resource.

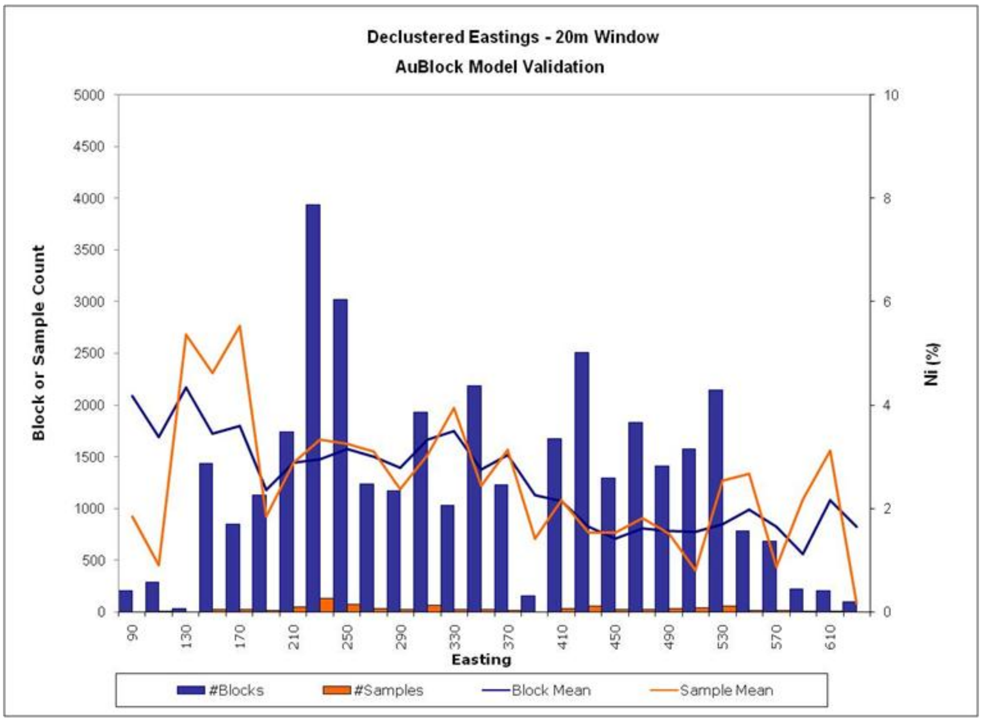


Figure 31. Validation plot for 20m Window Declustered Easting – MSV

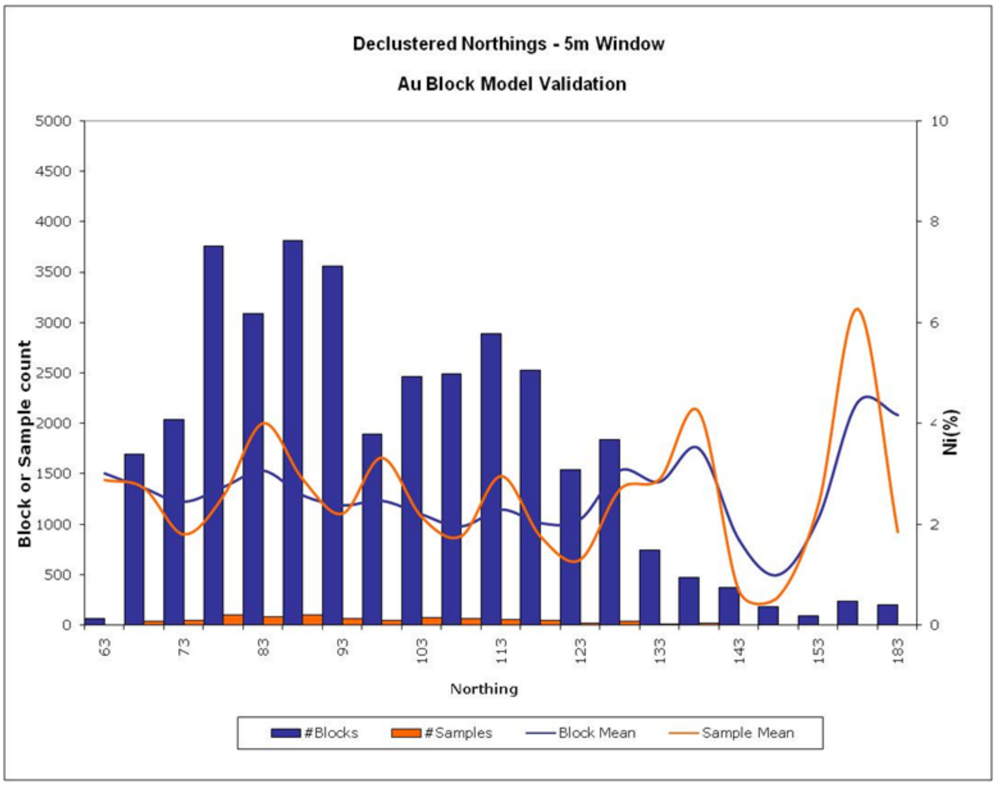


Figure 32. Validation plot for 5m Window Declustered Northing – MSV

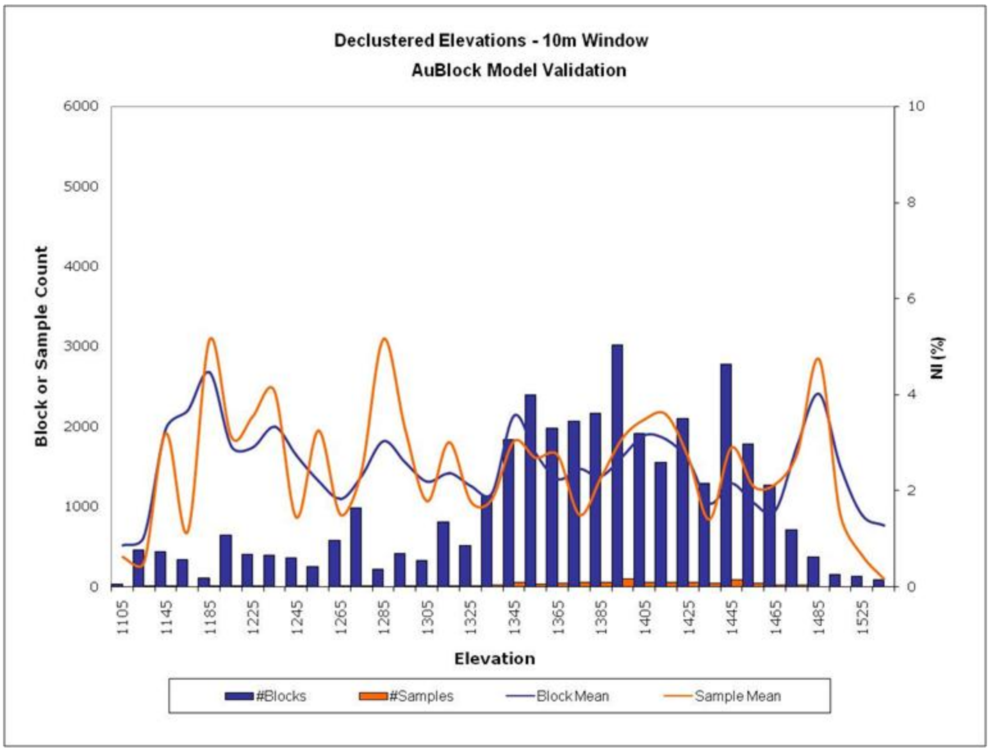


Figure 33. Validation plot for 10m Window Declustered Elevations - MSV.

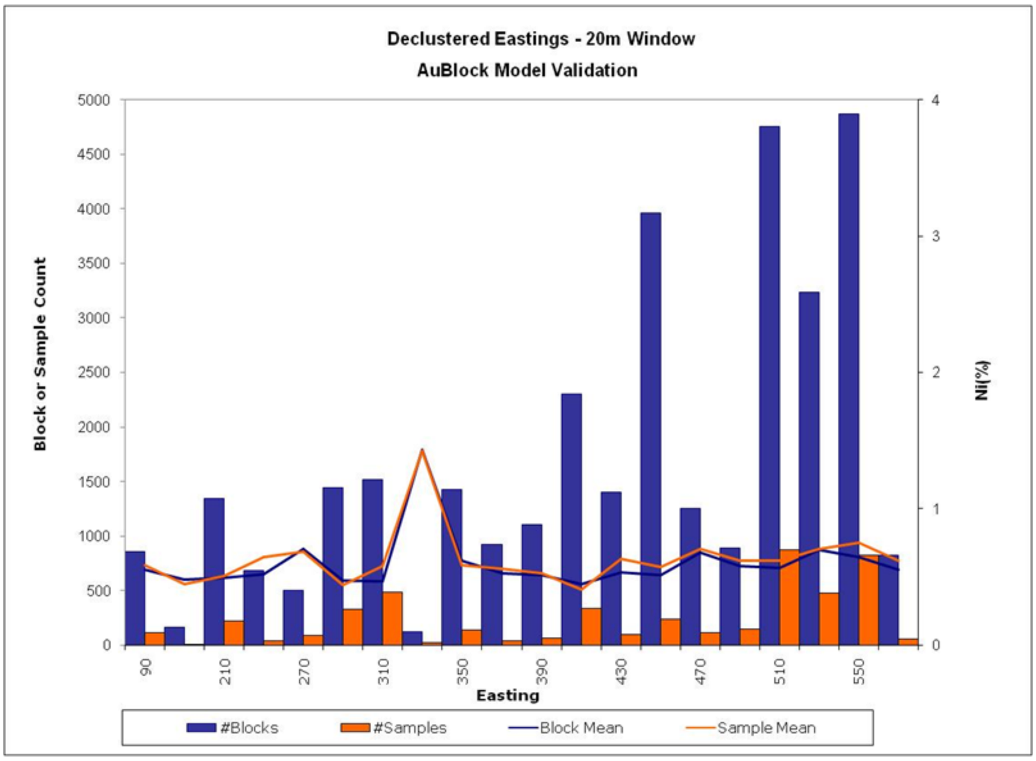


Figure 34. Validation plot for 20m Window Declustered Easting – DISS

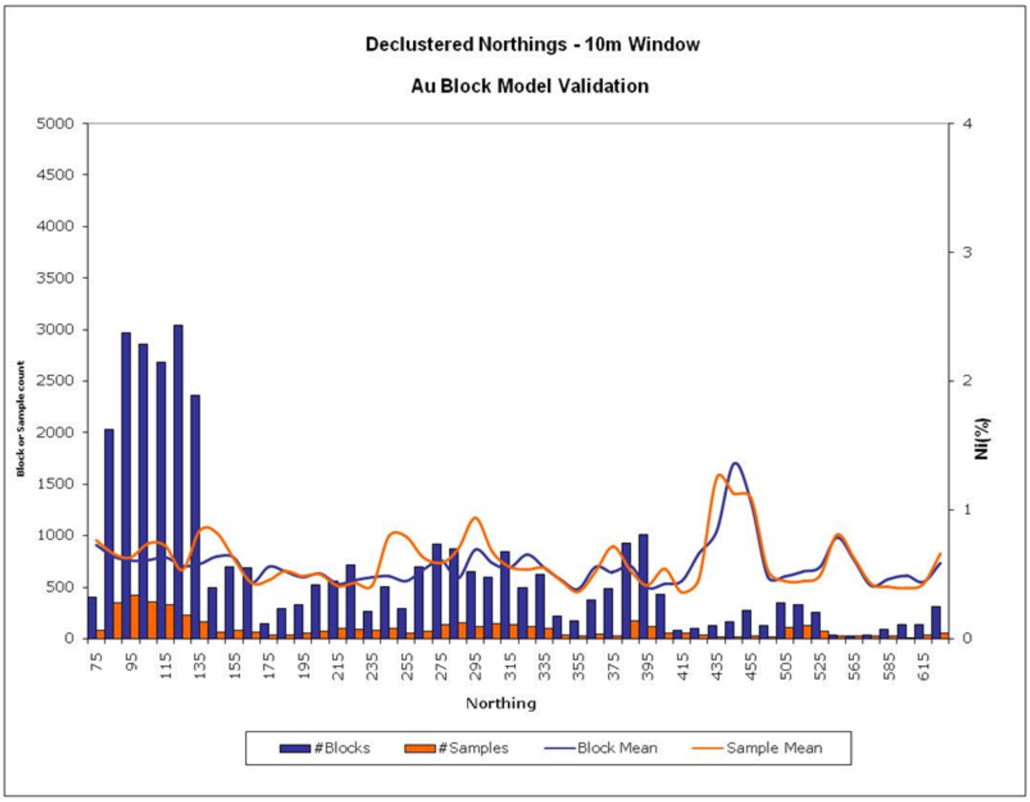


Figure 35. Validation plot for 5m Window Declustered Northing – DISS

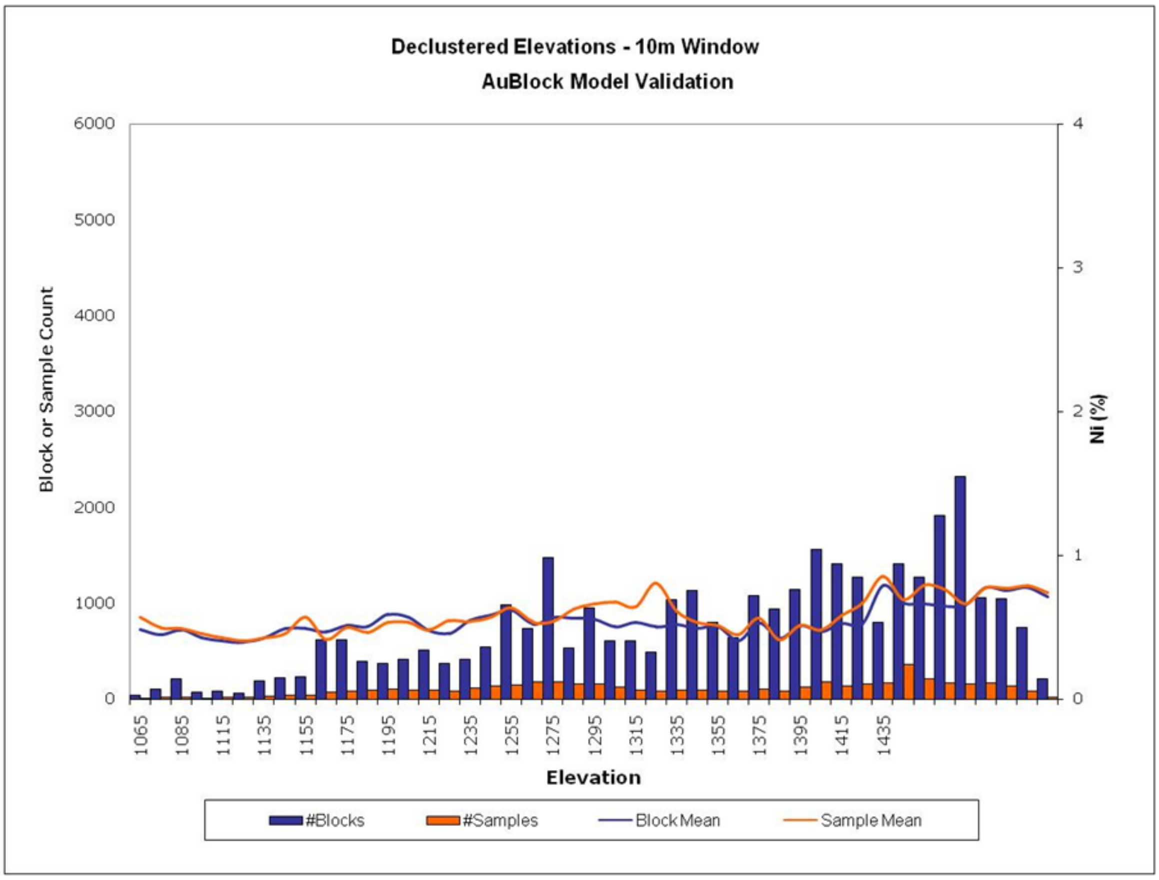


Figure 36. Validation plot for 10m Window Declustered Elevations – DISS

14.9.3 Smoothing Effects of Grade Interpolation

Estimated average block grades produced by linear interpolation methods (such as Inverse Distance and Ordinary Block Kriging) may introduce some degree of smoothing of grade and consequent conditional bias, i.e. the over-estimation of low grades and under-estimation of high grades. Ideally, the interpolation procedure should honour the contact between the mineable ore and the surrounding lower grade mineralization or waste rock. This would avoid smoothing of the grade across boundaries, providing more realistic estimates.

14.10 SG Estimate

SG was estimated by Ordinary Kriging (OK) method with other variables into blocks based on the database provided by BPNM to estimate metal tonnage by the accumulation method. As a check, a density field was also created and calculated by regression which showed no discernible difference.

The statistical analysis for SG and Density in the Block Model shows they have a strong correlation coefficient and very close mean values (See Figure 37).

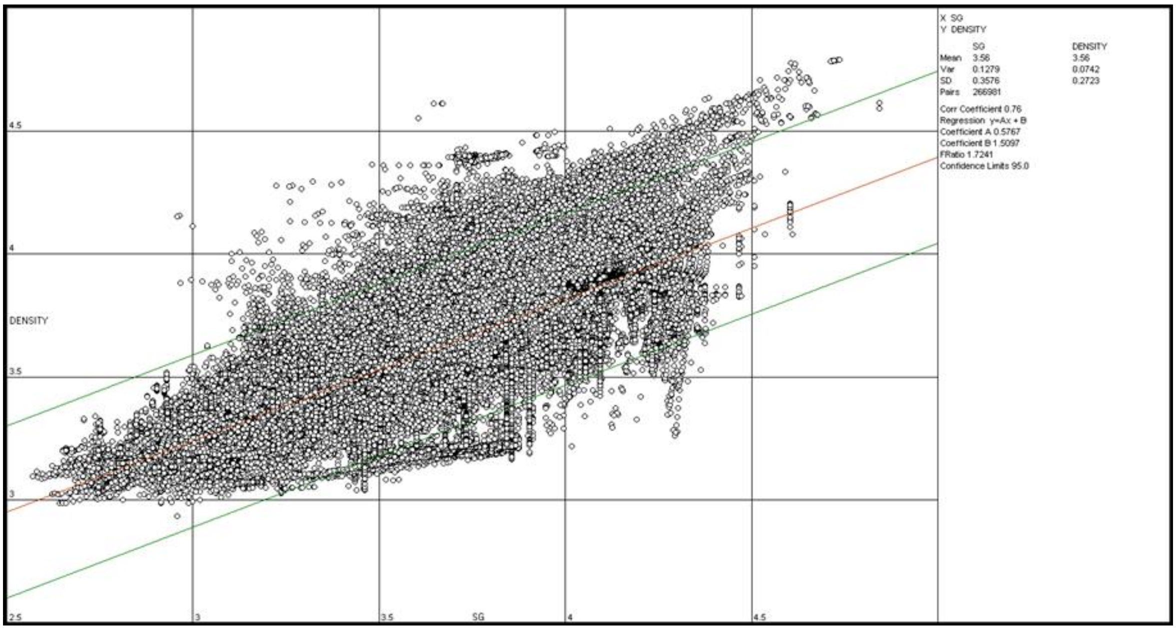


Figure 37. Correlation coefficient plot for SG Kriged and Density calculated

14.11 3m Waste Skin Expanded Block Model

As BPNM required, a 3m skin expanded waste block model was created and kriged by the method above without high-grade treatment. A "Full" block model was created by adding the waste block model to the mineralized model. The full block model can be used for the MSV mining dilution calculation.

14.12 Mineral Resource Classification

14.12.1 Philosophy

The Ban Phuc MSV Mineral Resource have been classified and reported in accordance with The 2004 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code). Resource classification is based on confidence in the geological domaining, drill spacing and geostatistical measures.

The initial classification process was based on an interpolation distance and minimum samples within the search ellipse as defined by the Micromine macro. The main components of the macro are summarized as follows:

Initial classification:

- The resource was classed as Inferred if the average weighted sample distance was greater than 60m.
- The resource was classed as Indicated if the average weighted sample distance was between 30m and 60m.
- The resource was classed as Measured if the average weighted sample distance was less than 30m.
- Numbers of drill holes -< 2m Measured and Indicated Resources downgraded one class.

The initial classification was reviewed visually. Based on the initial classification, three solids rescat_meas, rescat_ind and rescat_inf were created to define Measured, Indicated and Inferred Resources. This defined resource categories based on a combination of data density and geological confidence.

The resource classification codes in the model are as follows:

Measured Resource	(class = 1)
Indicated Resource	(class = 2)
Inferred Resource	(class = 3)
Unclassified Resource	(class = 4)

A range of criteria has been considered in determining the classification including:

- Geological continuity;
- Data quality;
- Drillhole spacing;
- Modelling technique; and
- Estimation parameters including search strategy, number of samples, average distance to samples to blocks and relative Kriging variance.

14.12.2 Data Quality

Mineral Resource classification is based on data collected and stored in the central BPNM database, provided to CSA for the resource estimation. It is considered that drilling techniques, survey, sampling and sample preparation, analytical techniques and database management and validation are well within industry standards.

14.12.3 Drill Spacing

Drill hole location plots have been utilized to ensure the drilling spacing meets the expected minimum requirements for resource classification. Measured material is defined where drilling is typically 25m by 25m and 25m by 30m spaced, and where lode continuity confidence is high. Indicated material is defined where drilling is typically 30m by 50m spaced, and where lode continuity confidence is high. Inferred material lies beyond the indicated boundaries and within the wireframe domains.

14.12.4 Modelling Technique

A conventional three dimensional (3D) Ordinary Kriging modelling technique has been used, with an unfolding methodology applied to provide a dynamic element to the allocation of search ellipses. The modelling technique is suitable to the domains being estimated allowing reasonable expectation of mining selectivity across the mineralized domain.

14.12.5 Estimation Properties

Information from the estimation process, including search pass, number of composites used in the search ellipse and Kriging variance are all used in conjunction with drill spacing to finalise classification domains. The final block classification is shown below in Figure 38 to Figure 40.

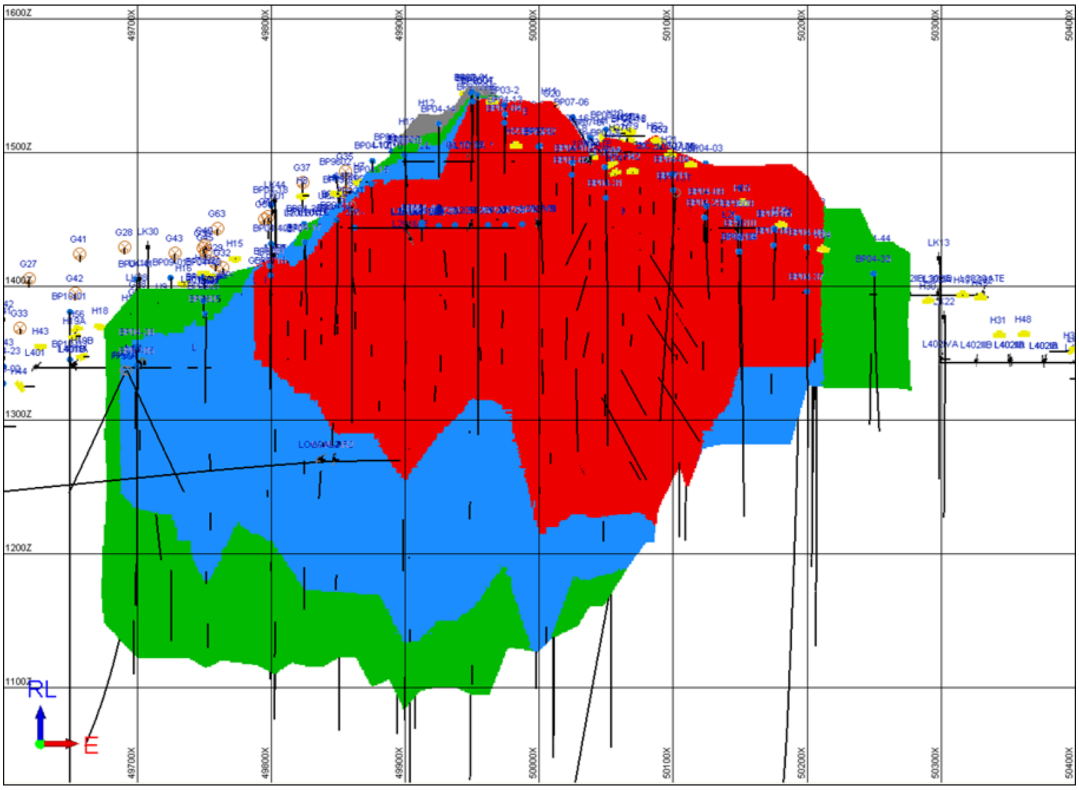


Figure 38. Looking North View on Ban Phuc MSV Resource Model Classification

(red=Measured; blue=Indicated; green=Inferred; grey=unclassified)

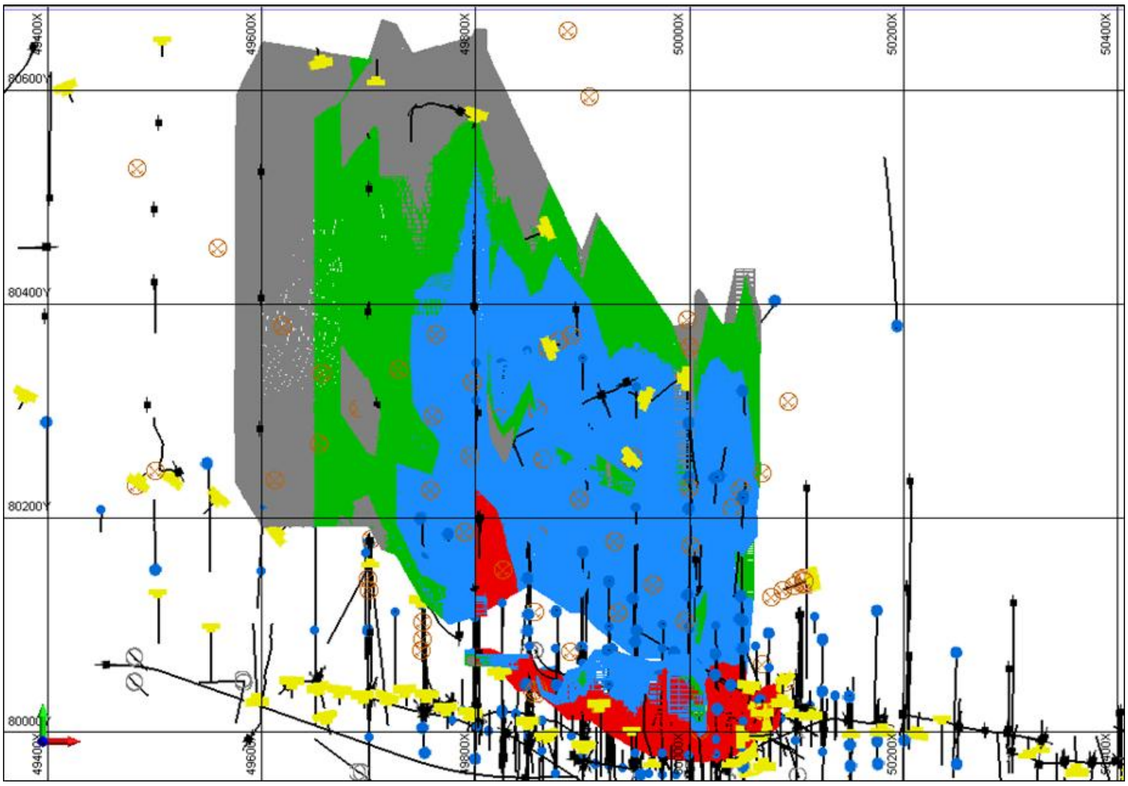


Figure 39. Plan View on Ban Phuc DISS Resource Model Classification

(red=Measured; blue=Indicated; green=Inferred; grey=unclassified)

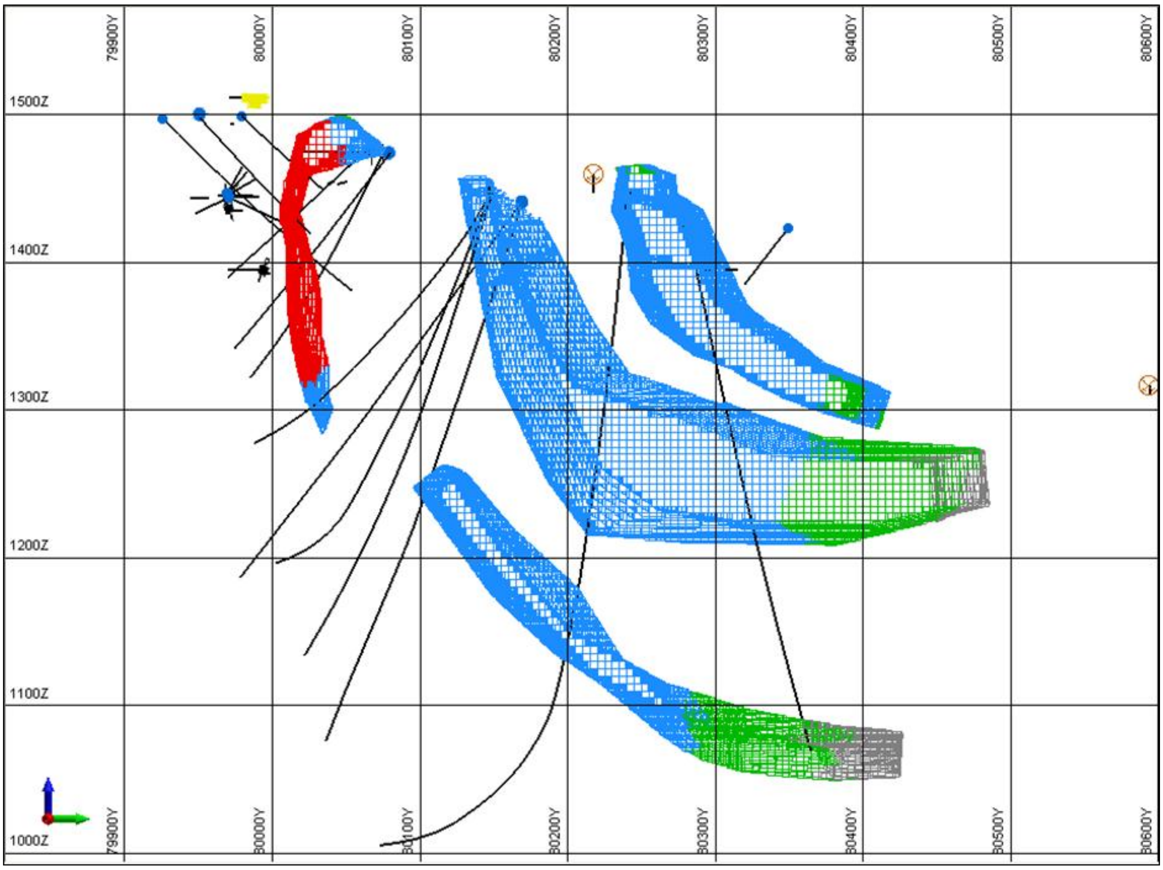


Figure 40. Section (49905E) view on Ban Phuc DISS Resource Model Classification

(red=Measured; blue=Indicated; green=Inferred; grey=unclassified)

14.13 Mineral Resource Reporting

14.13.1 Mining Depletion

There was no mining activity carried out in Ban Phuc Deposit at the time of carrying out the estimation, therefore there is no mining depletion incorporated into the model.

14.13.2 Cut- Off Grades

The current Ban Phuc MSV Mineral Resource has been reported above a cut-off of 0.40% Ni which considered by CSA to be a reasonable lower cut-off grade and is in accordance with industry standards.

The Ban Phuc DISS has been reported above a cut-off grade of 0.9%Ni. These cut-off grades have been determined in consultation with the site geologists and BPNM cost assumptions.

14.13.3 Mineral Resource Tables

A full summary of the resources for MSV and DISS are shown in Table 49 and Table 50.

Table 49. Mineral Resource estimate results for Ban Phuc MSV Deposit

Ban Phuc MSV Grade Tonnage Reported above a Cut off of 0.40% Nickel										
Category	**Tonnes (MT)**	**Ni Grade (%)**	**Cu Grade (%)**	**Co Grade (%)**	**S Grade (%)**	**MgO Grade (%)**	**Fe Grade (%)**	**Nickel (000'T)**	**Copper (000'T)**	**Cobalt (000'T)**
Measured	0.73	2.78	1.16	0.07	13.53	4.39	26.09	20	8	1
Indicated	0.96	2.60	1.22	0.06	12.94	2.04	25.01	25	12	1
Measured + Indicated	1.69	2.68	1.19	0.06	13.20	3.06	25.48	45	20	1
Inferred	0.17	1.94	0.80	0.03	10.04	6.76	20.27	3	1	0

***Note**: The CSA Mineral Resource was estimated within constraining wireframe solids based on a nominal lower cut-off grade of 0.4% Ni. Ordinary Kriging with high grade treatment. The resource is quoted from blocks above the specified Ni % cut-off grade.*

Table 50. Mineral Resource estimate results for Ban Phuc DISS Deposit

Ban Phuc DISS Grade Tonnage Reported above a Cut off of 0.90% Nickel										
Category	**Tonnes (MT)**	**Ni Grade (%)**	**Cu Grade (%)**	**Co Grade (%)**	**S Grade (%)**	**MgO Grade (%)**	**Fe Grade (%)**	**Nickel (000'T)**	**Copper (000'T)**	**Cobalt (000'T)**
Measured	0.2	1.05	0.15	0.01	1.14	15.83	3.75	2.1	0.3	0.0
Indicated	0.7	1.23	0.14	0.02	0.53	21.69	5.58	8.4	1.0	0.1
Measured + Indicated	**0.9**	**1.19**	**0.14**	**0.02**	**0.67**	**20.37**	**5.17**	**10.5**	**1.3**	**0.1**
Inferred	0.4	1.14	0.04	0.00	0.09	5.93	1.66	4.4	0.2	0.0

14.13.4 Grade Tonnage Tables

Detailed resource tabulations and grade tonnage curves for MSV are presented in the following tables (Table 51 and Table 52).

Table 51. Ban Phuc MSV Measured and Indicated Mineral Resource Grade and Tonnage Tabulations

Cut-off (Ni%)	Tonnes	Grade (Ni%)
0	1,691,311	2.68
0.2	1,691,258	2.68
0.4	1,691,247	2.68
0.6	1,690,847	2.68
0.8	1,686,566	2.69
1	1,669,647	2.70

Cut-off (Ni%)	Tonnes	Grade (Ni%)
1.2	1,634,268	2.74
1.4	1,555,129	2.81
1.6	1,459,283	2.90
1.8	1,340,254	3.00
2	1,234,851	3.10
2.2	1,131,092	3.19
2.4	993,500	3.31
2.6	854,238	3.44
2.8	705,667	3.60
3	564,896	3.77
3.2	459,590	3.93
3.5	332,899	4.15
3.8	203,824	4.46
4	159,690	4.62
4.2	119,247	4.79
4.5	73,448	5.07
4.8	41,365	5.40
5	32,164	5.54
5.2	21,770	5.75
5.5	11,781	6.10
6	3,503	6.92

Table 52. Ban Phuc MSV Indicated Mineral Resource Grade and Tonnage Tabulations

Cut-off (Ni%)	Tonnes	Grade (Ni%)
0	958,230	2.60
0.2	958,230	2.60
0.4	958,223	2.60
0.6	957,903	2.60
0.8	956,263	2.61
1	948,761	2.62
1.2	928,986	2.65
1.4	875,547	2.73
1.6	817,260	2.82
1.8	739,824	2.94
2	681,145	3.03
2.2	629,682	3.11
2.4	546,290	3.23
2.6	452,580	3.38
2.8	360,918	3.55
3	276,242	3.75

Cut-off (Ni%)	Tonnes	Grade (Ni%)
3.2	225,001	3.90
3.5	163,659	4.09
3.8	87,679	4.47
4	68,979	4.63
4.2	53,801	4.78
4.5	34,976	5.02
4.8	17,301	5.37
5	14,484	5.47
5.2	8,857	5.69
5.5	5,796	5.89
6	559	6.86

Detailed resource tabulations and grade tonnage curves for DISS are presented in the following tables (Table 53 and Table 54).

Table 53 Ban Phuc DISS Measured and Indicated Mineral Resource Grade and Tonnage Tabulations

DISS Measured and Indicated in-situ Resource		
Cut-off (Ni%)	**Tonnes**	**Grade (Ni%)**
0	20,525,152	0.50
0.1	20,525,152	0.50
0.2	20,525,152	0.50
0.3	20,178,788	0.50
0.4	12,475,711	0.59
0.5	7,024,102	0.71
0.6	4,169,546	0.82
0.7	2,632,179	0.92
0.8	1,584,999	1.04
0.9	887,765	1.19
1	562,980	1.33
1.2	287,628	1.57
1.4	158,164	1.80
1.6	113,855	1.92
1.8	104,406	1.94
2	645	2.09

Table 54. Ban Phuc DISS Inferred Resource Grade and Tonnage Tabulations

DISS Inferred in-situ Resource		
Cut-off (Ni%)	**Tonnes**	**Grade (Ni%)**
0	11,035,123	0.50
0.1	11,035,123	0.50
0.2	11,035,123	0.50
0.3	11,001,310	0.50
0.4	7,415,440	0.56
0.5	3,902,822	0.67
0.6	2,006,440	0.79
0.7	1,070,860	0.92
0.8	662,102	1.02
0.9	387,798	1.14
1	250,611	1.25
1.2	102,613	1.47
1.4	38,157	1.79
1.6	25,465	1.96
1.8	25,465	1.96

15 Mineral Reserve Estimate

AMDAD prepared a Mineral Reserve estimate for underground mining by "up-hole retreat benching, without backfill". The approach taken to prepare the estimate, based on geological and resource data provided by CSA, is outlined below.

The MSV lens has been re-interpreted by CSA since an initial mine plan was prepared by AMDAD in 2007, and following an update of the mine plan by AMDAD in 2009/2010. The interpreted lens is narrow in places, i.e. 1.2 m but the widest zones are approximately 9m to 10m.

For the 2009/2010 mine plan, AMDAD used a 3.0 m minimum mining width extraction wireframe constructed by a local consultant for BPNM using Surpac software. In an effort to reduce dilution and increase the mining grade, 2.5 m wide extraction wireframes were constructed by AMDAD for the new mine plan based on the lens wireframes supplied by CSA. The Figure 41 to Figure 43 below show the 2.5 m minimum mining width wireframes, which do not include the near surface oxide zones.

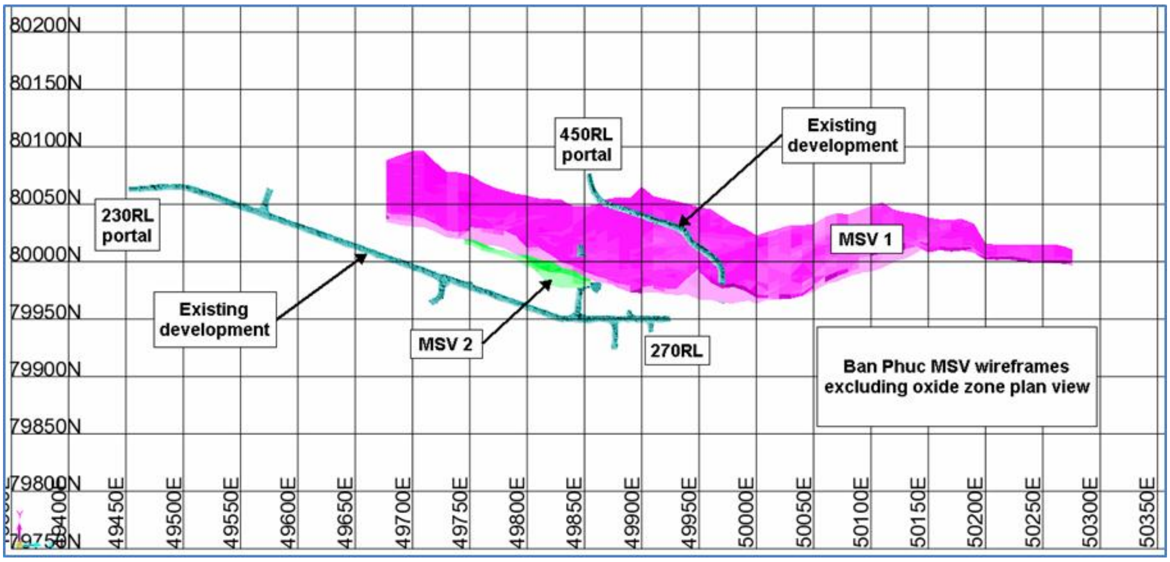


Figure 41. Ban Phuc MSV wireframes (plan view)

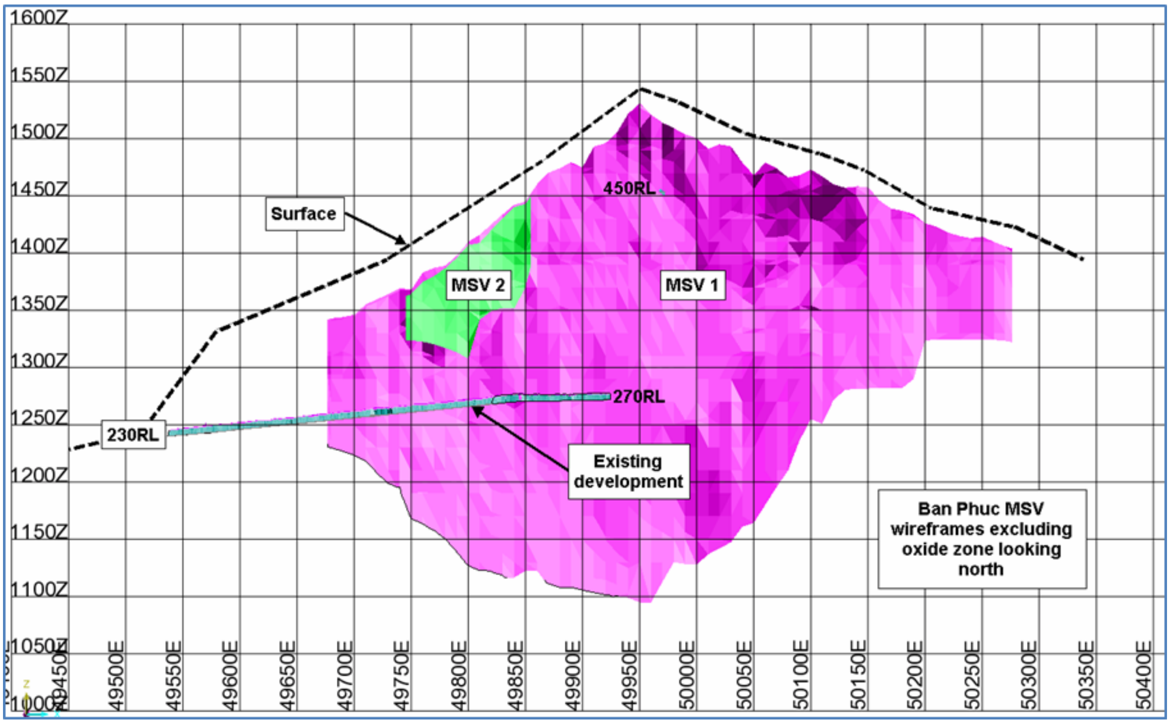


Figure 42. Ban Phuc MSV 2.5m wireframes for Lode 1 and Lode 2 (looking north)

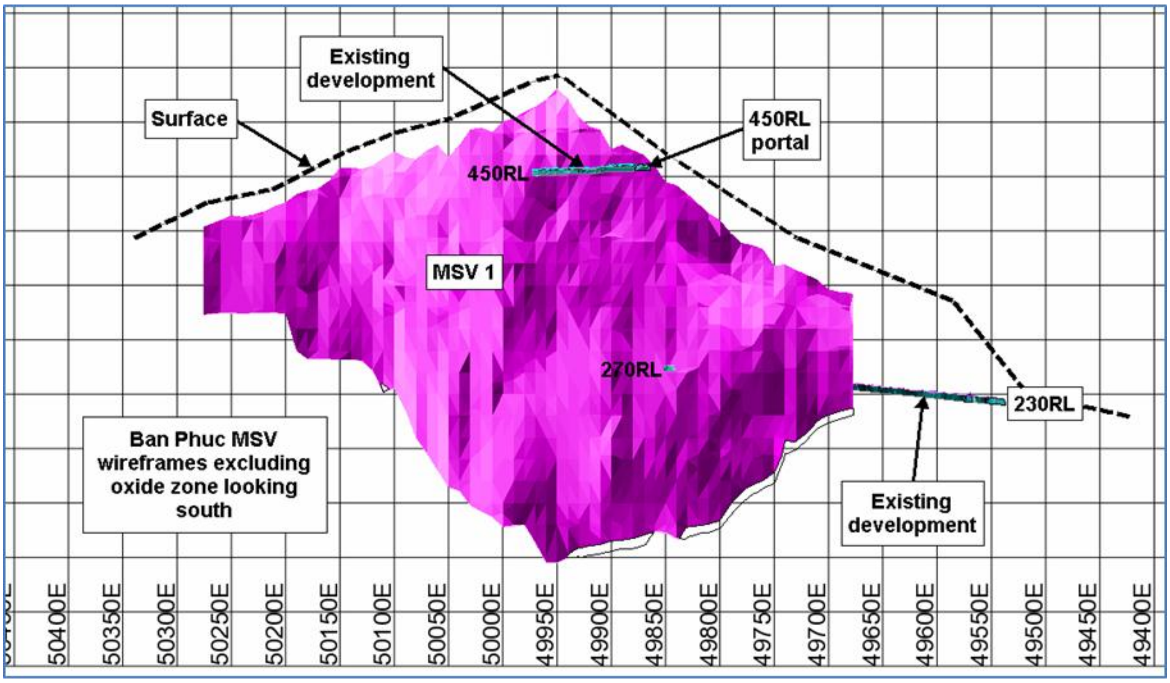


Figure 43. Ban Phuc MSV wireframes (looking south)

After application of the minimum mining-width to the resource, processing and economic factors were applied as follows. The processing and economic parameters in Table 55, provided by BPNM, were used to determine nickel-equivalence factors and a nickel-equivalent cut-off head grade for the selected mining method.

Nickel equivalent (NiEq) grade = nickel grade + (copper grade x 0.27) + (cobalt grade x 0.59).

Table 55. Cut-off grade parameters

Item	Unit	Value
Nickel metallurgical recovery	%	90
Copper metallurgical recovery	%	95
Cobalt metallurgical recovery	%	75
Nickel price	US$/t	21,319
Copper price	US$/t	8,419
Cobalt price	US$/t	34,000
Concentrate Haulage	US$/wmt	77.2
Sea Freight	US$/wmt	21
Conc. Grade Control	US$/wmt	2.2
Amount Payable, Nickel	%	72
Amount Payable, Copper	%	50
Amount Payable, Cobalt	%	30
Tariff	%	10
Employee wages & benefits	US$/t	24.26
Mining Costs - development	US$/t	35.28
Processing	US$/t	17.64
Administration	US$/t	5.51
Contingency	US$/t	7.72
Cut-off Grade, NiEq - design	%	0.86
NiEq factor, Copper		0.27
NiEq factor, Cobalt		0.59

Operating cost assumptions shown were escalated by AMDAD by a factor of 10.25% to reflect increases between 2010 and 2012.

The mining parameters presented in Table 56 and Table 57 were applied to the mining width adjusted-resource to define tonnes and grades for practical 20 m x 20 m mining blocks. Only those blocks, from Measured and Indicated Resources, with diluted NiEq grade above the cut-off grade were included in the Mineral Reserves.

Table 56. Mineral Reserve parameters

Item	Unit	Value	Comment
Nickel (equivalent) cut-off grade	%	0.86	
Minimum 20 x 20 block size	tonnes	1,400	This removes all the small, narrow sections of the 2.5m wireframe (at the edges)
Minimum development width	m	4	This will allow for efficient mucking in the development sills
Maximum development width	m	6	Any width > 6m will be mined as part of the bench.
Development height	m	4.5	To allow sufficient space for the production drill, vent bags etc.
Minimum pillar size	m^2	40	
Stope block length	m	20	
Stope block height	m	15.5	
Crown pillar thickness	m	7.8	Vertical thickness of pillar, therefore 7.7m of bench is mined in crown pillar benches
Stope dilution, Ni	%	Varies	Grade reported between the 2.5m wireframe and a 4.5m wireframe (the 2.5m wireframe was expanded by a metre on both sides) is used as the dilution grade for each 20 x 20m block
Stope dilution, Cu	%		
Stope dilution, Co	%		

Table 57. Dilution and recovery assumptions for Up-hole Retreat Benching

Item	Unit	Development	Production	>1430RL	>1350RL	<1350RL
Dilution	%			10%	5%	4%
General Mining recovery	%	100%	95%			

Crown pillars have been assigned to the following levels; 430, 310 and 210 RLs.

As an example, the following figures demonstrate the effects of these factors. Figure 44 shows all the blocks inside the 2.5 m MSV1 and MSV2 wireframes colour coded by block tonnes. Figure 45 indicates how many blocks would be included in the Mineral Reserve once the parameters and cut-off grade are applied.

LODE1 wireframes

Insitu, 2.5m WF 20 x 15.5 blk Tonnes

RL	TONNES	49650	49670	49690	49710	49730	49750	49770	49790	49810	49830	49850	49870	49890	49910	49930	49950
1530																	
1510	3,119															1,220	1,899
1490	10,732														399	2,993	3,675
1470	23,770												396	1,442	3,516	3,232	3,663
1450	44,732											1,104	4,053	5,201	2,745	2,713	4,421
1430	57,458										1,729	3,794	4,462	5,259	3,387	2,581	2,932
1410	72,654								10	3,309	4,522	4,627	5,196	4,953	3,079	2,947	3,381
1390	88,789							63	2,864	3,992	3,550	5,251	4,807	5,551	4,409	3,538	3,335
1370	97,264						1,371	3,605	2,795	2,908	4,110	4,099	4,071	4,292	4,195	3,529	3,355
1350	100,320			4	1,453	3,762	2,697	3,681	4,046	3,069	4,642	4,072	3,382	2,707	2,687	2,841	2,700
1330	112,082		1,725	4,219	6,193	4,763	3,353	4,058	9,384	4,587	5,288	4,490	3,757	2,574	2,463	2,291	2,302
1310	98,237		3,188	2,833	4,409	4,365	3,499	4,293	5,909	4,666	5,408	5,214	3,116	2,612	2,529	2,336	2,458
1290	84,597		1,696	2,230	3,143	4,112	2,828	4,627	3,459	3,769	4,850	5,240	3,474	3,037	2,601	2,269	2,588
1270	73,948		1,456	2,364	3,104	4,146	2,338	3,165	2,770	3,032	4,399	4,326	3,505	3,506	2,894	2,256	3,106
1250	67,099		1,593	2,521	2,861	3,951	2,287	2,750	2,775	3,200	3,365	3,700	4,259	4,140	3,139	2,666	2,823
1230	68,903		1,667	2,778	3,239	3,870	2,618	2,631	3,495	3,330	3,856	4,174	4,577	4,487	3,871	3,458	3,192
1210	72,965		1,672	2,843	3,151	3,747	2,881	3,191	4,682	3,800	4,029	4,545	5,678	4,935	4,208	3,580	3,300
1190	74,661		1,497	2,467	2,676	3,355	3,801	3,591	6,172	4,153	4,369	4,732	6,078	4,812	4,312	3,372	3,277
1170	72,684		1,514	2,346	2,771	4,156	5,304	4,048	5,194	3,601	4,013	4,801	5,504	4,981	3,846	3,362	3,168
1150	69,864		1,510	2,321	2,943	3,522	5,007	4,819	4,205	3,760	3,964	4,561	5,837	5,274	4,541	2,972	3,180
1130	63,000		719	2,238	2,975	3,176	3,146	3,416	3,187	3,804	4,100	4,450	6,224	5,734	5,199	3,181	3,158
1110	44,388			703	1,348	1,898	1,761	1,647	2,559	3,321	3,089	2,977	5,790	5,971	5,799	3,565	2,909
1090																	
	1,401,266																

RL	49970	49990	50010	50030	50050	50070	50090	50110	50130	50150	50170	50190	50210	50230	50250	50270
1530																
1510																
1490	2,809	840	16													
1470	3,528	3,223	3,327	1,059	384											
1450	3,026	2,492	2,541	3,028	4,059	2,327	4,348	2,046	579	49						
1430	3,085	2,659	2,865	3,608	3,597	3,497	3,612	4,390	3,668	2,046	287					
1410	2,667	3,223	3,301	2,393	2,389	2,847	5,216	6,541	4,083	2,545	2,930	1,920	562	13		
1390	3,011	3,554	3,065	2,114	2,528	3,581	5,829	7,346	4,652	2,759	2,888	2,780	2,466	2,392	1,956	508
1370	2,783	3,720	4,104	2,655	2,634	4,607	5,712	8,482	5,162	3,440	3,270	3,598	2,852	2,664	2,416	835
1350	3,297	5,433	3,806	3,189	2,955	3,972	4,354	6,643	4,138	4,137	4,186	3,805	3,250	2,318	2,285	809
1330	2,864	4,948	2,578	3,243	2,413	2,481	3,181	4,179	3,540	4,578	5,449	3,028	2,773	2,387	2,302	691
1310	2,909	3,082	2,753	3,980	2,625	2,109	3,392	4,468	3,030	3,681	5,110	1,562	834	798	757	312
1290	3,377	2,542	3,384	3,138	2,241	2,088	3,076	3,954	2,847	3,061	4,364	602				
1270	3,821	3,197	2,934	2,749	2,000	2,305	3,039	3,142	1,230	1,409	1,755					
1250	3,793	4,027	2,585	2,652	1,930	2,291	2,666	1,125								
1230	3,572	3,059	3,443	2,962	2,168	2,157	299									
1210	3,027	2,801	3,461	3,523	2,280	1,631										
1190	3,530	2,713	3,763	3,340	2,183	468										
1170	3,451	2,537	3,336	3,468	1,283											
1150	3,311	2,680	2,999	2,381	77											
1130	3,576	3,441	1,264	12												
1110	806	245														
1090																

MSV 1 lens (block tonnes)

LODE2 wireframes

Insitu, 2.5m WF 20 x 15.5 blk Tonnes

RL	TONNES	49650	49670	49690	49710	49730	49750	49770	49790	49810	49830	49850	49870	49890	49910
1530															
1510															
1490															
1470															
1450															
1430	2,040										1,383	657			
1410	12,359									2,240	9,258	861			
1390	22,879							14	2,588	9,065	9,436	1,776			
1370	17,897						306	2,230	3,926	4,214	6,225	996			
1350	18,499					446	2,751	2,473	4,396	3,780	4,653				
1330	9,874					459	2,123	2,585	4,348	359					
1310	4,400					83	717	1,718	1,882						
1290	11								11						
1270															
1250															
1230															
1210															
1190															
1170															
1150															
1130															
1110															
1090															
	87,959														

MSV 2 lens (block tonnes)

Figure 44. 20m x 20m block layout in 2.5m mining wireframes showing block tonnes.

LODE2 wireframes

Economic Stopes, tonnes

Elevation	TONNES	49650	49670	49690	49710	49730	49750	49770	49790	49810	49830	49850	49870	49890	49910
1530															
1510															
1490															
1470															
1450															
1430	1,521										1,521				
1410	12,073									2,352	9,721				
1390	24,097								2,717	9,518	9,908	1,954			
1370	17,425							2,342	4,122	4,425	6,536				
1350	18,956						2,889	2,597	4,616	3,969	4,886				
1330	9,418						2,208	2,688	4,522						
1310	4,140							1,976	2,164						
1290															
1270															
1250															
1230															
1210															
1190															
1170															
1150															
1130															
1110															
1090															
	87,630														

MSV 2 lens (economic stope tonnes)

LODE1 wireframes

Economic Stopes, tonnes

Elevation	TONNES	49650	49670	49690	49710	49730	49750	49770	49790	49810	49830	49850	49870	49890
1530														
1510	3,587													
1490	10,425													
1470	24,124													1,586
1450	47,300												4,458	5,721
1430	62,888										1,902	4,173	4,908	5,785
1410	75,989									3,474	4,748	4,858	5,456	5,201
1390	85,818								3,007	4,192	3,728	5,514	5,047	5,829
1370	93,266						1,440	3,785	2,935	3,053	4,316	4,304	4,275	4,507
1350	96,514				1,526	3,950	2,832	3,865	4,248	3,222	4,874	4,276	3,551	2,842
1330	108,362		1,794	4,388	6,441	4,954	3,487	4,220	9,759	4,770	5,500	4,670	3,907	2,677
1310	100,313		3,347	2,975	4,629	4,583	3,674	4,508	6,204	4,899	5,678	5,475	3,272	2,743
1290	87,646		1,764	2,319	3,269	4,276	2,941	4,812	3,597	3,920	5,044	5,450	3,613	3,158
1270	75,948		1,514	2,459	3,228	4,312	2,432	3,292	2,881	3,153	4,575	4,499	3,645	3,646
1250	69,091		1,657	2,622	2,975	4,109	2,378	2,860	2,886	3,328	3,500	4,070	4,685	4,306
1230	71,873		1,734	2,889	3,369	4,025	2,723	2,736	3,635	3,463	4,010	4,591	5,035	4,666
1210	76,613		1,756	2,985	3,309	3,934	3,025	3,351	4,916	3,990	4,230	4,772	5,962	5,182
1190	73,982			2,566	2,783	3,489	3,953	3,735	6,419	4,319	4,544	5,205	6,686	5,004
1170	69,694			2,440	2,882	4,322	5,516	4,210	5,402	3,745	4,174	5,281	6,054	5,180
1150	62,593			2,414	3,061	3,663	5,207	5,012	4,373	3,910	4,123	5,017	6,421	5,485
1130	36,611					3,303	3,272	3,553	3,314	3,956	4,264			5,963
1110	6,762								2,943	3,819				
1090														
	1,339,399													

Elevation	49910	49930	49950	49970	49990	50010	50030	50050	50070	50090	50110	50130
1530												
1510		1,403	2,184									
1490		3,292	4,043	3,090								
1470	3,868	3,555	4,029	3,881	3,545	3,660						
1450	3,020	2,984	4,863	3,329	2,741	2,795	3,331	4,465	2,560	4,783	2,251	
1430	3,726	2,839	3,225	3,394	2,925	3,152	3,969	3,957	3,847	3,973	4,829	4,035
1410	3,233	3,242	3,719	2,800	3,384	3,466	2,513	2,508	2,989	5,477	6,868	4,287
1390	4,629	3,892	3,669	3,162	3,732	3,218	2,220	2,654	3,760	6,120	7,713	4,885
1370	4,405	3,882	3,691	2,922	3,906	4,309	2,788	2,766	4,837	5,998	8,906	5,420
1350	2,821	3,125	2,970	3,462	5,705	3,996	3,348	3,103	4,171	4,572	6,975	4,345
1330	2,562	2,520	2,532	2,979	5,146	2,681	3,373	2,510	2,580	3,308	4,346	3,682
1310	2,655	2,453	2,581	3,054	3,236	2,891	4,179	2,756	2,214	3,562	4,691	3,182
1290	2,705	2,496	2,847	3,512	2,644	3,519	3,264	2,331	2,172	3,199	4,112	2,961
1270	3,010	2,482	3,417	3,974	3,325	3,051	2,859	2,080	2,397	3,161	3,268	
1250	3,265	2,773	2,936	3,945	4,188	2,688	2,758	2,007	2,383	2,773		
1230	4,026	3,596	3,320	3,715	3,181	3,581	3,080	2,255	2,243			
1210	4,418	3,759	3,465	3,178	2,941	3,634	3,699	2,394	1,713			
1190	4,484	3,507	3,408	3,671	2,822	3,914	3,474					
1170	4,000	3,496	3,295	3,589	2,638	3,469						
1150	4,723	3,091	3,307		2,787							
1130	5,407				3,579							
1110												
1090												

Elevation	50150	50170	50190	50210	50230	50250	50270	50290	50310	50330	50350
1530											
1510											
1490											
1470											
1450											
1430	2,251										
1410	2,672	3,077	2,016								
1390	2,897	3,032	2,919								
1370	3,612	3,434	3,778								
1350	4,344	4,395	3,995								
1330	4,761	5,667	3,149								
1310	3,865	5,366	1,640								
1290	3,183	4,539									
1270	1,465	1,825									
1250											
1230											
1210											
1190											
1170											
1150											
1130											
1110											
1090											

MSV 1 lens (economic stope tonnes)

Figure 45. Mineral Reserve blocks in 2.5m mining wireframe

The resultant estimated Mineral Reserves after application of these parameters is tabulated in Table 58.

Table 58. Estimated Mineral Reserve

Item	Mt	Ni grade %	Cu grade %	Co grade %
Proven Mineral Reserve	0.71	2.4	1.0	0.06
Probable Mineral Reserve	0.9	2.1	1.0	0.04
Total Mineral Reserve	1.6	2.2	1.0	0.05

***NB**: This Estimated is consistent with the Mineral Reserves stated in July 2010.*

16 Mining Methods

16.1 Introduction

Data supplied to AMDAD includes:

1. Block models from CSA, model_ok_full_final.dm and model_ok_msv_final.dm

2. Geological wireframes, representing the MSV zone bp_msv_lode1.dtm (lens 1) and bp_msv_lode2.dtm (lens 2), provided by BPNM

3. Survey pickups for the mining development on 270 and 450 RLs provided by BPNM

4. Pells, Sullivan, Meynink (PSM) June 2009 report, "Stoping Design Review, Ban Phuc Nickel Mine"

5. PSM report PSM1166.R3 Draft, June 2010 "Geotechnical review of stoping, Ban Phuc"

6. Financial and cost estimates supplied by BPNM, in RK - Ban Phuc Handover Notes_MINING_Dec 2008.doc, which includes spreadsheets with cost estimates

The MSV lens has been re-interpreted by CSA since the initial mine plan was prepared by AMDAD in 2007 and following an update of the mine plan by AMDAD in 2009/2010. The interpreted lens is narrow in places, i.e. 1.2 m, but the widest zones are approximately 9 m to 10 m.

For the 2009/2010 mine plan, AMDAD used a 3.0 m minimum mining width extraction wireframe constructed by a local consultant for BPNM using Surpac software. In an effort to reduce dilution and increase the mining grade, 2.5 m wide extraction wireframes were constructed by AMDAD for the new mine plan based on the lens wireframes supplied by CSA. The figures below show the 2.5 m minimum mining width wireframes, which do not include the near surface oxide zone.

16.2 Mining Method

The selected mining method is "up-hole retreat benching, without backfill". This method was chosen because of the following factors:

- Simple mining method;
- Lower operating and capital cost method to those involving backfill;
- Top down method, which enables earlier access to ore; and
- With several benches in operation, the target production rate of 360,000tpa nominated by BPNM is considered achievable.

The following diagram (Figure 46) indicates how the method will be used at Ban Phuc.

Referring to the diagram, the method is as follows:

1. The orebody will be accessed via crosscuts developed from a central decline/incline system at 20 m vertical intervals.

2. 4.5 m high sill drives or "sills" will be mined along the orebody from the crosscut on a 20m (floor to floor) vertical spacing.

3. The sills will be mined to 4.0 m minimum width and to a maximum of 6.0 m, ground conditions permitting. The orebody width ranges from 2.5 m to 9 m, for 2.5 mWF MSV1 and 2.5 m to 10 m for the 2.5 mWF MSV2.

4. Starting from the far end, either east or west, the orebody will be drilled out with nominal 64 mm diameter blastholes holes. The holes, approximately 16 m long (15.5 m vertical dimension) will be drilled upwards towards the next sublevel above.

5. A suitable production drill rig would drill an up-hole longhole rise ("LHR") and cut-off slot out to the width of the orebody.

6. Parallel holes will be drilled in "rings" back from the LHR, to the extent of the predicted stable span. To aid in charging and firing these rings will be angled upwards toward the LHR.

7. This will form a drilled out panel. This pattern of LHR and rings is repeated back to the access crosscut.

8. Once production of the lift above has been completed in that area, the up-hole LHR can be fired. Firing of rings would follow as required with the size of the blasts tailored to suit the ground conditions in the bench.

9. Load-haul-dump ("LHD") units will load or "muck" the ore out of the stope. Mining the sills to a 4 m minimum width should enable ease of mucking.

10. Conventional mucking could be used whilst the brow is closed.

11. Once the brow is opened, it is proposed that tele-remote mucking be used. Pillars of various sizes, as shown in the figure above, will be left in between the panels to stabilize the walls of the void, as the void is not filled.

12. The sizes of crown pillars in between vertically adjacent production regions were set as recommended by PSM.

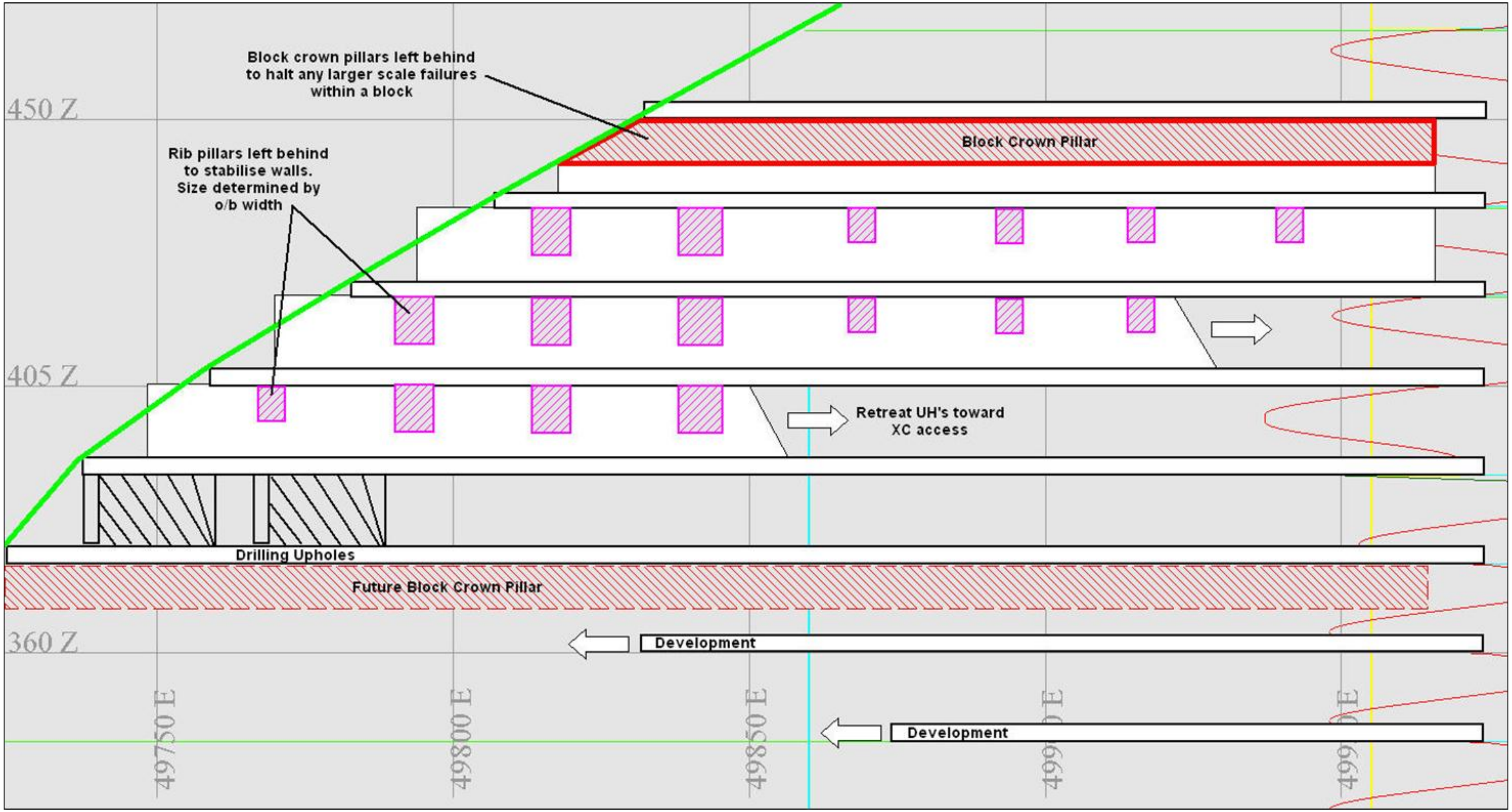


Figure 46. Up-hole Retreat Benching

13. The production rate of each panel - a group of benches between crown pillars - depends largely upon the width of the orebody; wide areas yield higher tonnes per metre of strike, so mucking in between firings will be more continuous. Narrow areas may yield only enough fired ore for part of a shift, after which charging and firing would need to be undertaken again.

14. The use of crown/sill pillars means that any failures of stope walls or backs will be arrested by the crown/sill pillar left at the top of the next panel. This will eliminate dilution from previously mined areas progressing to the next panel, although at the expense of lower ore recovery due to more ore being left in pillars.

16.3 Geotechnical

Pells Sullivan Meynink (PSM) conducted a review of the geotechnical aspects of the mining including:

- Review of geology data;
- Review of mining method and mine design;
- Numerical analysis of a section along the centreline of the MSV; and
- Recommendations for geotechnical parameters.

Initially, PSM was provided with an early mining plan with 15 m level spacing, as the starting point for their evaluation. They were then asked to provide advice in relation to a mine plan with 20 m vertical level spacing. Both these mine plans used a minimum mining width of 3 m.

PSM was then asked to evaluate the new mine plan with 2.5 m minimum mining width, 20 m vertical level spacing, and updated wireframe by CSA. The key findings of their review, as applicable to a 20 m sub-interval mine design are listed below;

- A 20 m vertical stope spacing with discontinuous rib pillars every 20 m along strike is likely to require thickening of one or more crown and rib pillars and a reduction in stope strike length in some areas. The suggested crown pillar thickness is 7.8 m and rib pillar dimension is 40 m^2.
- The crown pillar for 20 m sub-interval layout will require a 30% increase in the crown pillar thickness. The resultant thickness is 6 m x 1.3 = 7.8 m.
- The rib pillars will need to be increased by 10%. Therefore the rib pillar area is 36 m^2 x 1.1 = 40 m^2.

16.4 Hydrological

Mine hydrology was investigated by Knight Piésold in conjunction with a 2005 Feasibility Study documented by Ausenco. Boreholes were drilled to assess hydrogeological conditions at the location for the underground mine. The boreholes were drilled with portable drill rigs using rotary HQ coring techniques using water as a drilling fluid. Packer permeability tests were carried out at regular intervals to assess the permeability of the rock foundations at the underground mine borehole location.

Based on the results of the hydrogeological investigation and analysis the following recommendations and conclusions were made:

- A maximum groundwater inflow of between 10 m^3/h (dry season) and 40 m^3/h (wet season) at the maximum mine development depth should be incorporated into the mine dewatering system. Underground mine waters may become acidic if in open contact with the massive sulphide material. Allowance has been made for neutralizing with lime.

- The impact of the underground mine on the Da River Reservoir (situated 4.5 km away) will be minimal as the minimum elevation of the mine workings is above the maximum elevation of the reservoir and will therefore not result in any drawdown effects. The groundwater flow from the mine area to the reservoir will be maintained but at a reduced rate. It is noted that groundwater flow from the mine area to the reservoir is currently insignificant (approximately 1,700 m^3/year reducing to 100 m^3/year).

16.5 Mine Design

A centreline design for mine development headings was constructed using Surpac software, guided by:

- 2.5m minimum mining width wireframe constructed by AMDAD;
- The current survey pickups of development; and
- Previous design work completed by AMDAD.

This mine development design was used for both development quantities and with the MineSched software to produce a development and production schedule. Figure 47 shows this design.

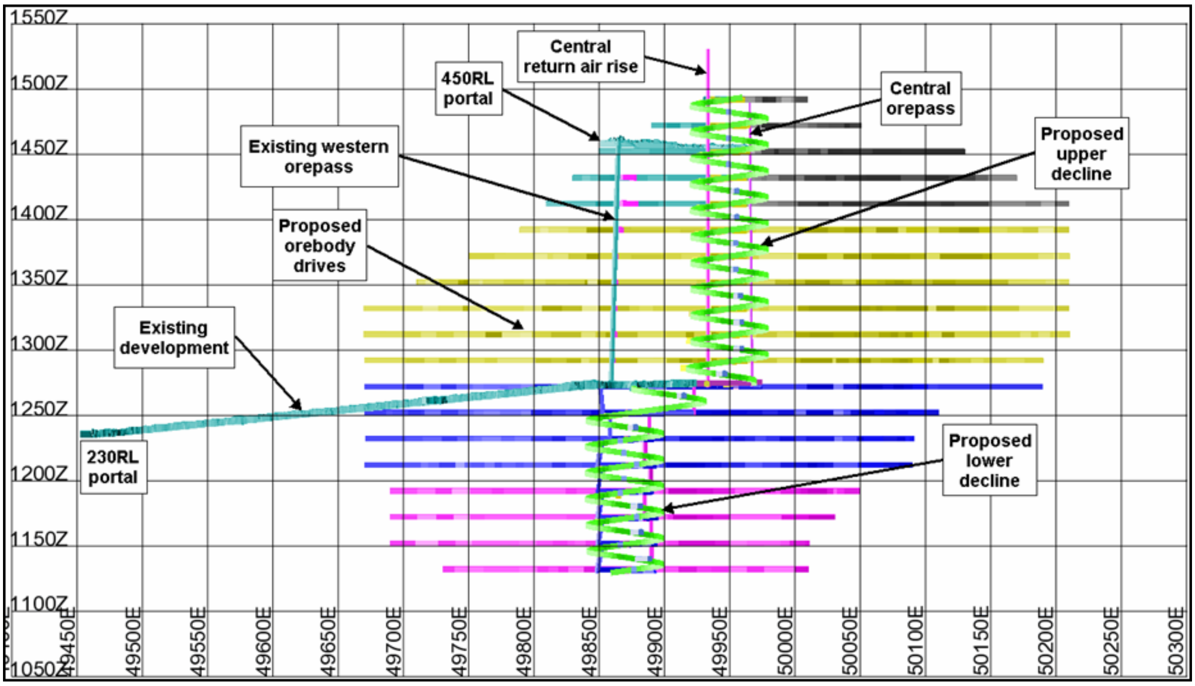


Figure 47. Mine design layout (looking north).

***NB**: this development was designed to suit the blocks that are included in the mining inventory. The declines have been positioned at central locations alongside the mineable resource above and below 270RL.*

With the crown pillars allocated to 430, 310 and 210 RLs, the mine is divided into 'Production Areas', whereby each Area can be mined independently of others to help achieve the required production rate. These 4 Areas are shown in Figure 54 below. The intention is to mine each block from the top bench down. Table 59 is a summary of the development metres for this design.

Table 59. Development Quantities

Development type	Metres
Decline development (waste)	2,791
Access development (waste)	1,095
Stockpile bays (waste)	449
Other (waste)	1,010
Sill development (ore) MSV1 east	4,083
Sill development (ore) MSV1 west	3,605
Sill development (ore) MSV2 east	406
Total Lateral development	13,439
Total Vertical development	723

The following figures (Figure 48 to Figure 51) show the layout of the mine for some of the levels, in plan view;

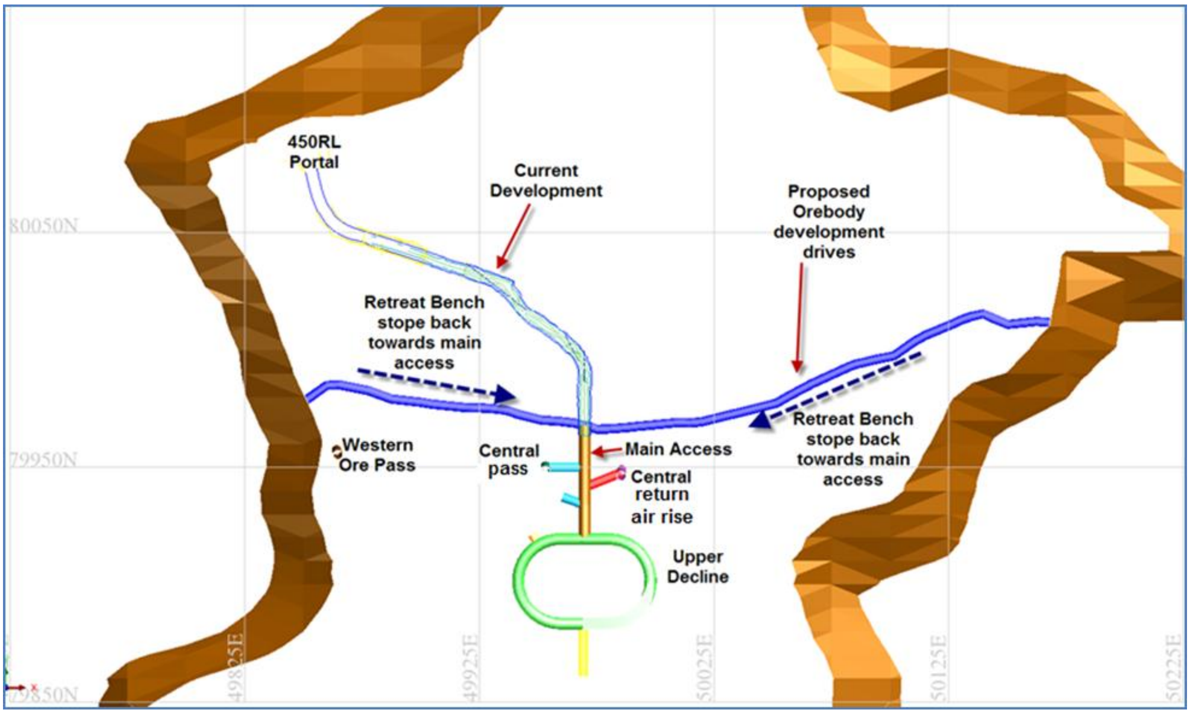


Figure 48. Mine design layout, 450RL

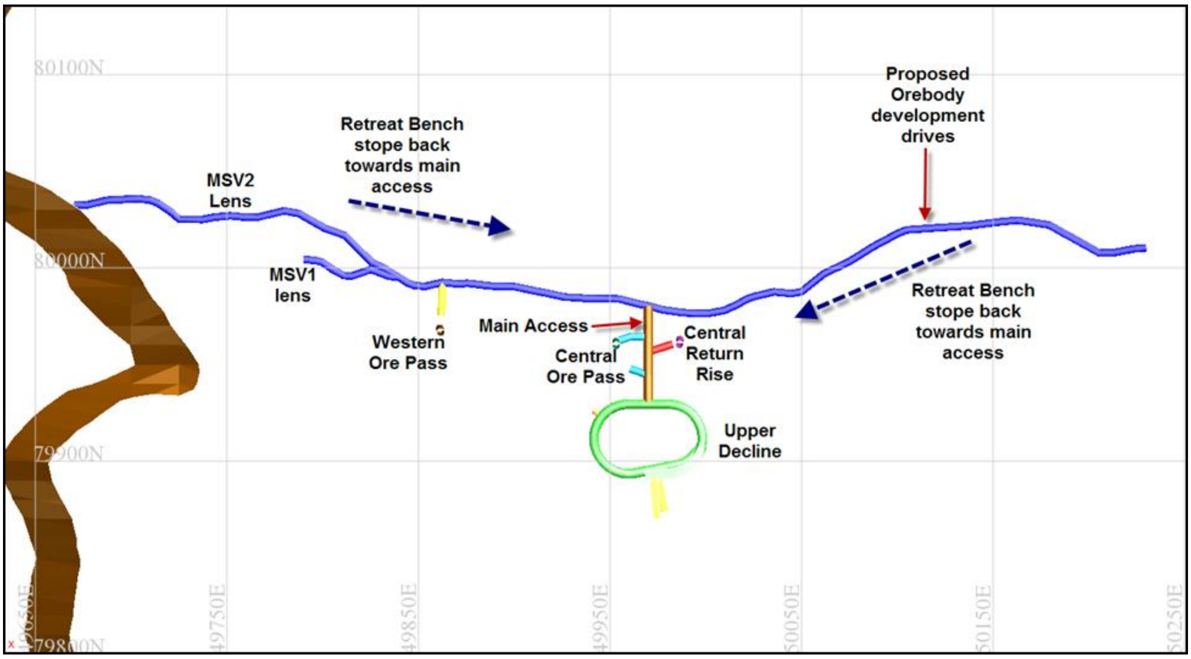


Figure 49. Mine design layout, 350RL

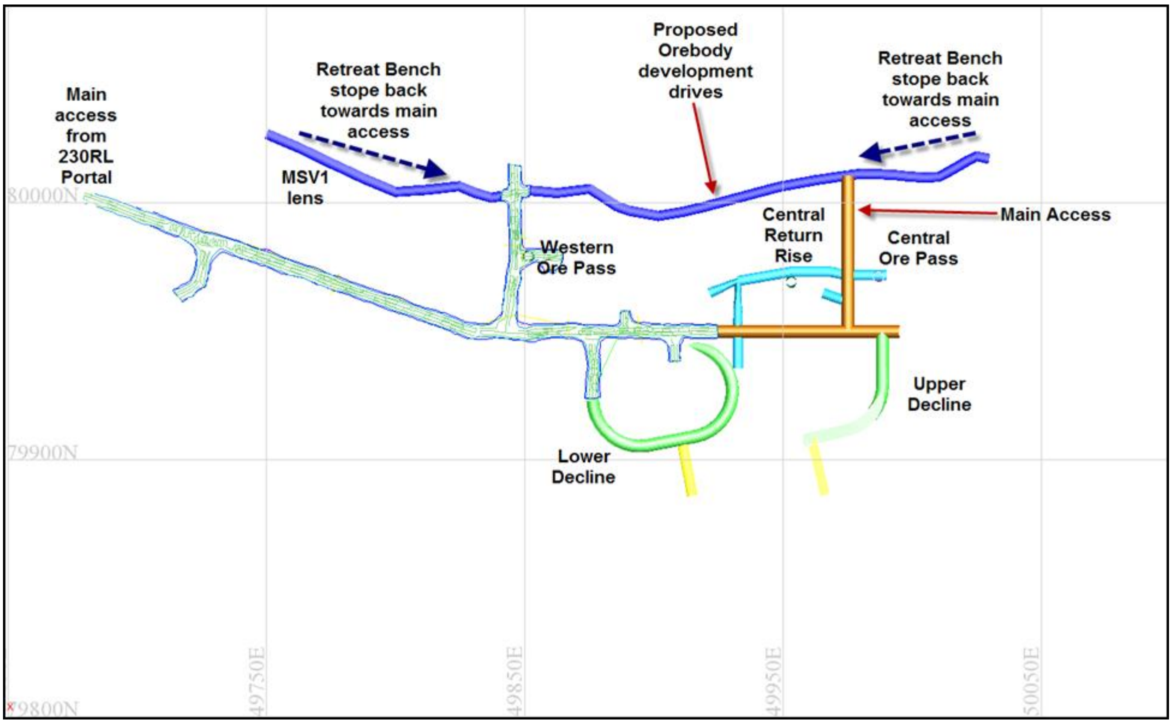


Figure 50. Mine design layout, 270RL

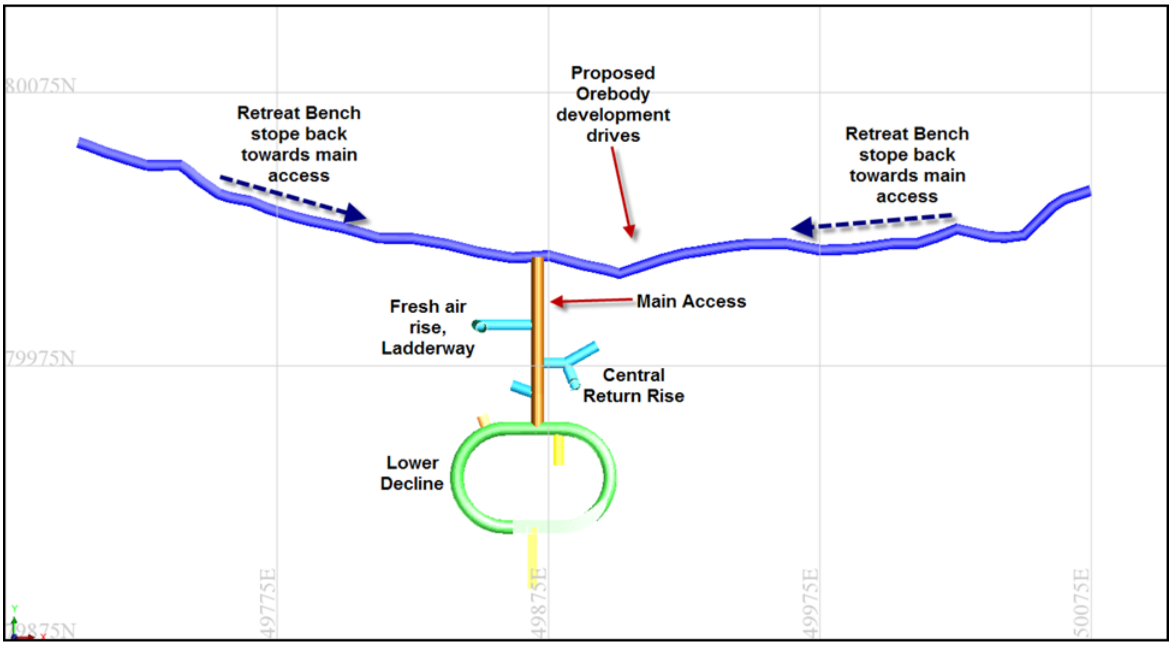


Figure 51. Mine design layout, 210RL

16.6 Mine Layout

Access to the mine is via two portals that have already been established. Development from each has been progressed to the MSV1 vein, as can be seen in the figures above. The Lower Portal, the main mine access, is located near the process plant area at 235RL (see Figure 52). The upper portal is at 450RL.

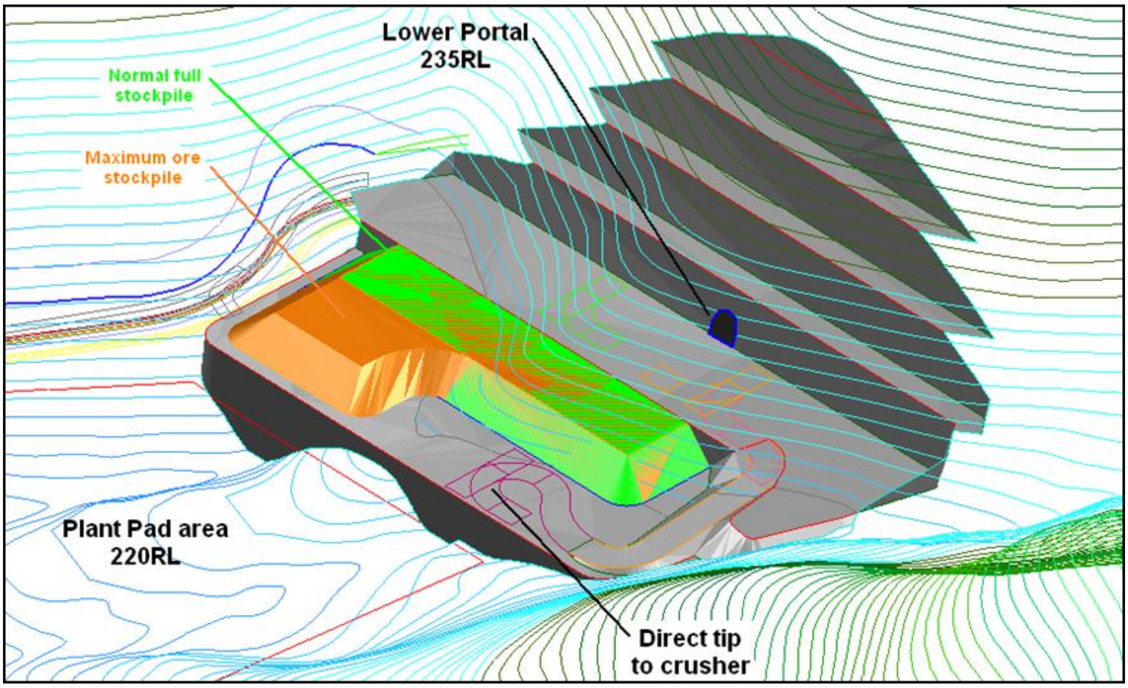


Figure 52. Lower Portal and Run-of-Mine Pad.

Fresh air will enter the mine via the two portals and will be exhausted initially through the Western Pass, before the Central Ventilation Rise is mined. Below 270RL, fresh air will be pulled down the decline and returned up a series of rises to 270RL, where it will be directed into the main ventilation rise to surface. A dedicated fresh air rise will also be continued down from 270 to 130RL to provide a second means of egress below 270RL.

All waste and ore above the 270RL level will be directed to the 270RL level via two passes, the Western Pass, already mined, and the Central Pass. Ore and waste below 270RL will be hauled up to 270RL. From the 270RL level, all rock will be hauled 900m to the ROM pad.

16.7 Mine Schedule

Figure 53 shows the four main Production Areas. These are then subdivided to the east and west of the declines. This gives eight production areas in MSV1 and one production area in MSV2.

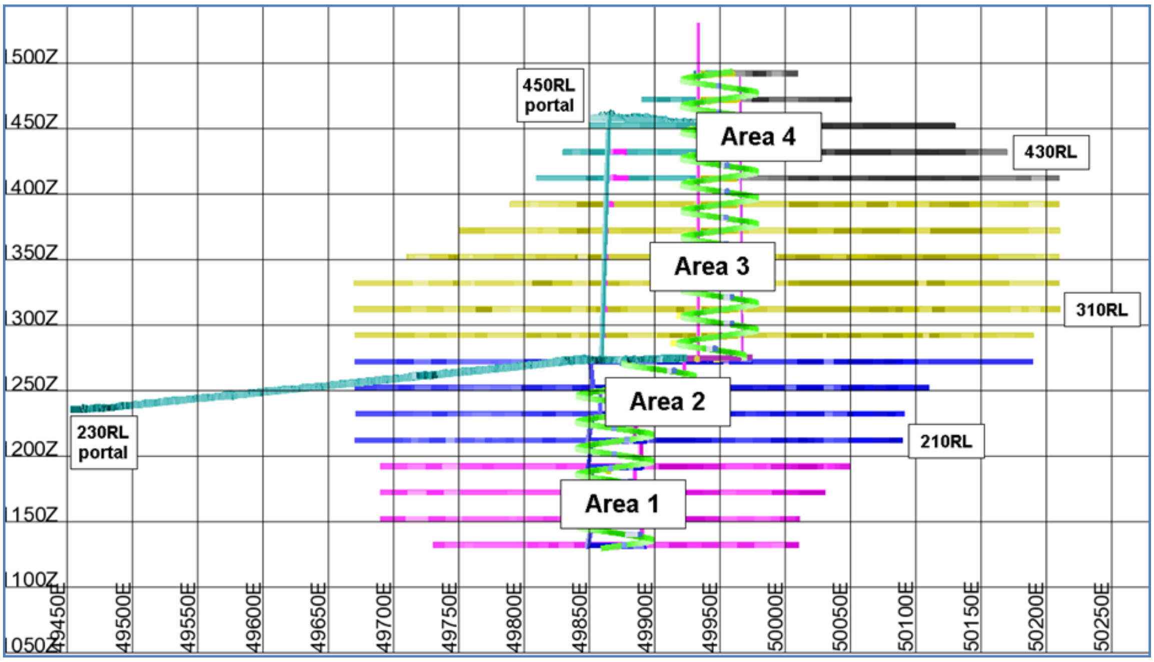


Figure 53. Mine design layout, showing Production Areas (looking north).

The development and production schedules are created in MineSched, so that iterations can be easily run for different development rates, prioritising different Areas and changing production rates.

The development schedule is based on following:

- Decline development targets 120 m/month in a single decline heading;
- Development in orebody sills range from 100 m to 160 m per month, depending on the width of the sill; and
- Total maximum development with two jumbos is 420 m per month and with one jumbo up to 200 m/month.

The production schedule is based on following:

- Maximum production from a single bench is 800 tonnes per day, when the source is available;
- Production cannot start in a bench until a lag of 15 days after development for that bench is complete to allow for survey, and for drilling of the first 20m stope; and
- Multiple benches can be in operation at once.

The development schedule is presented graphically in longitudinal projection (facing north) in the following figures (Figure 54 to Figure 63). The scheduled development and production quantities are presented in Table 60. Figure 64 displays monthly tonnes expected. Table 64 shows the estimate of required mining fleet.

AMDAD prepared a mining cost estimate corresponding to this schedule based on unit costs and factors supplied by BPNM escalated by 10.25%. This cost estimate is presented in Section 21.

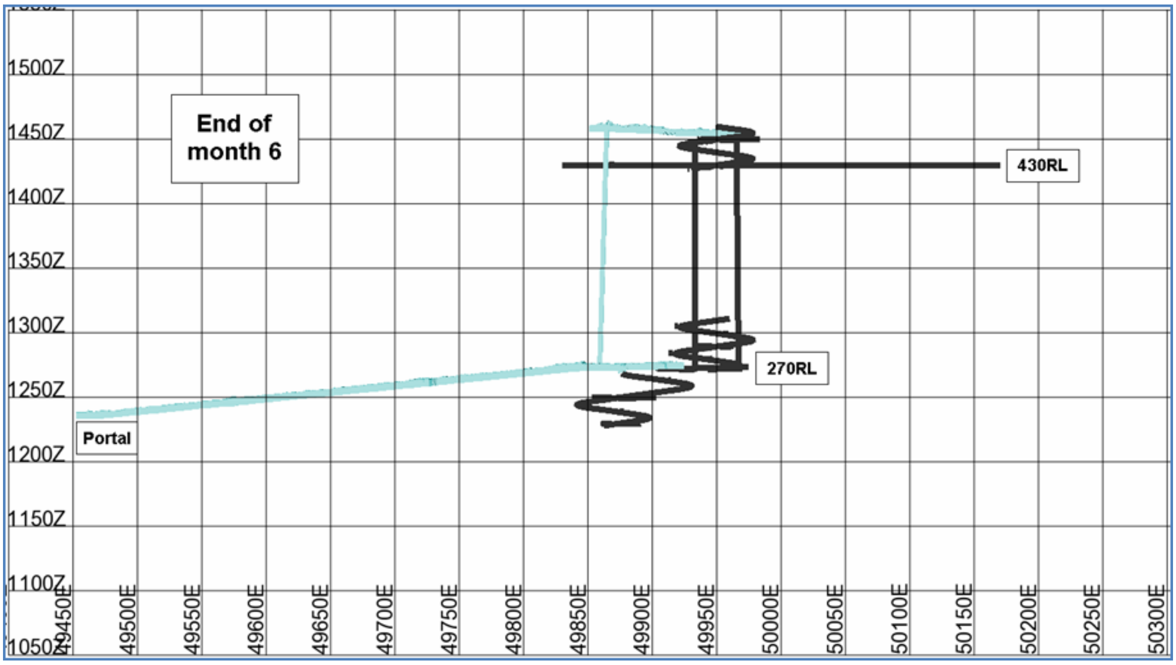


Figure 54. Development, end of Month 6.

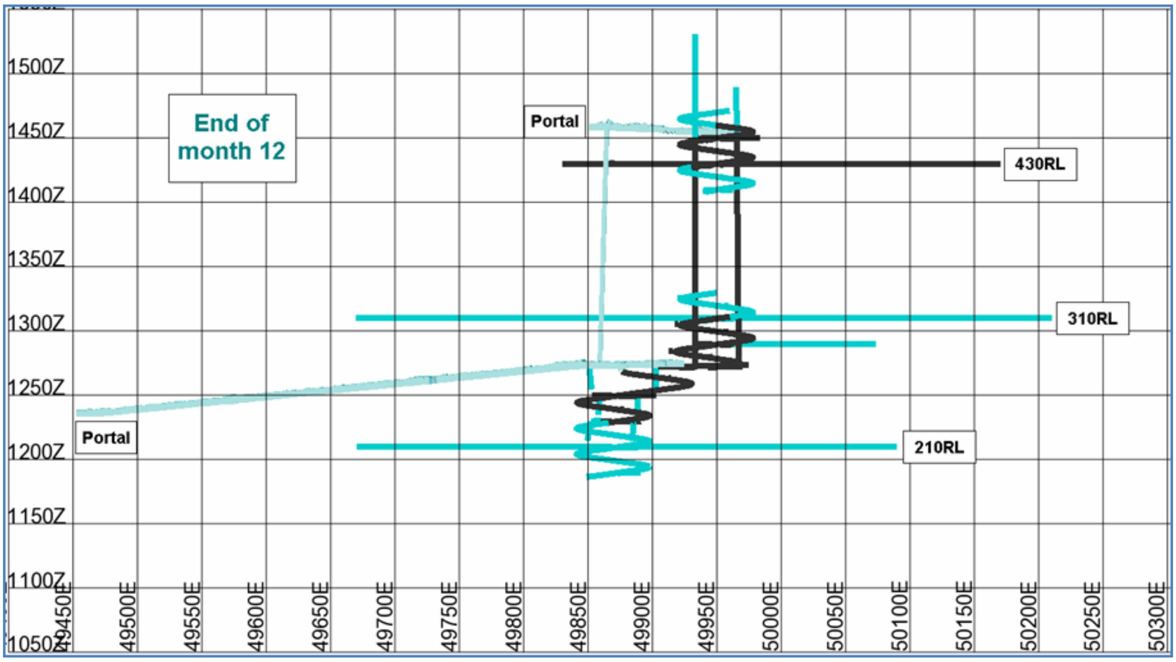


Figure 55. Development, end of Month 12.

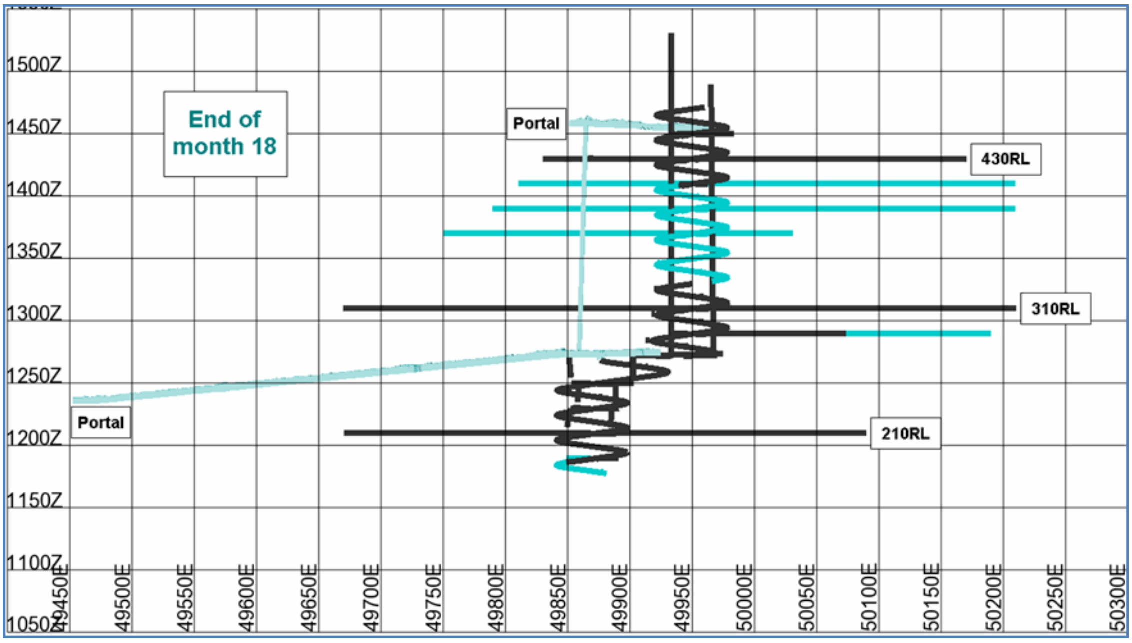


Figure 56. Development, end of Month 18.

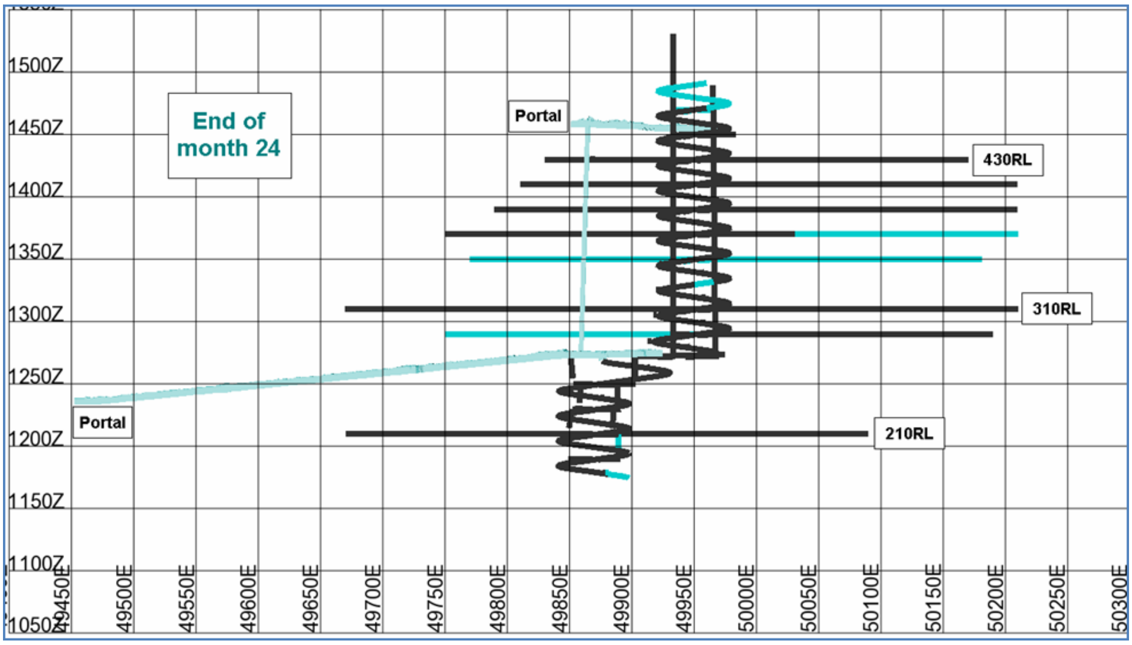


Figure 57. Development, end of Month 24.

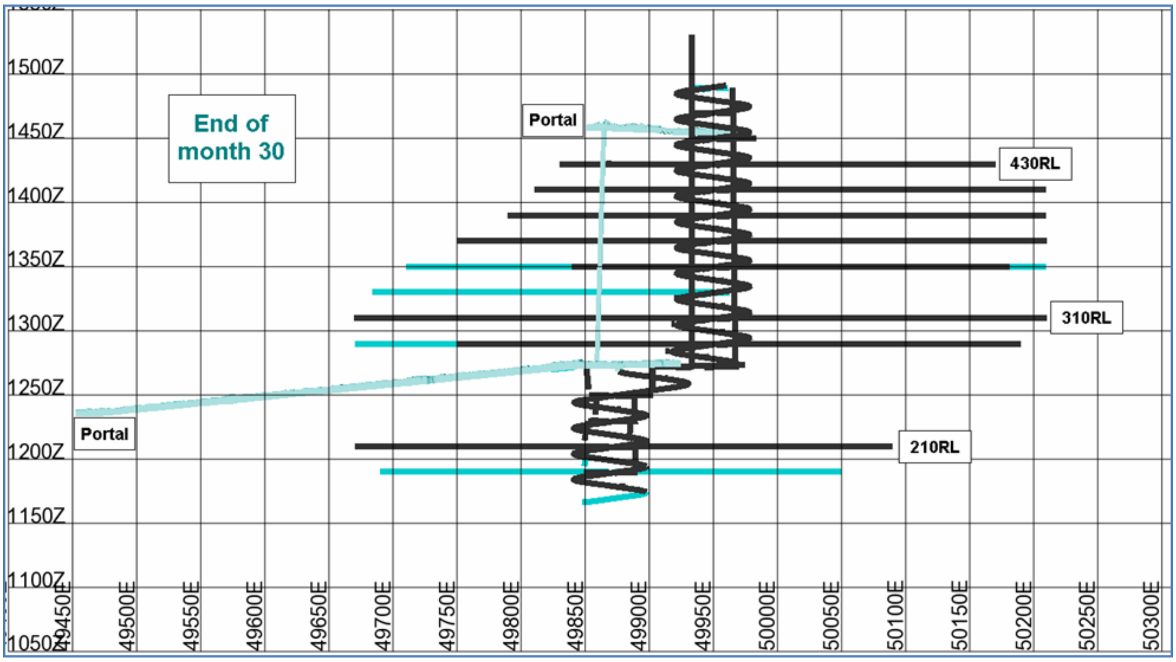


Figure 58. Development, end of Month 30.

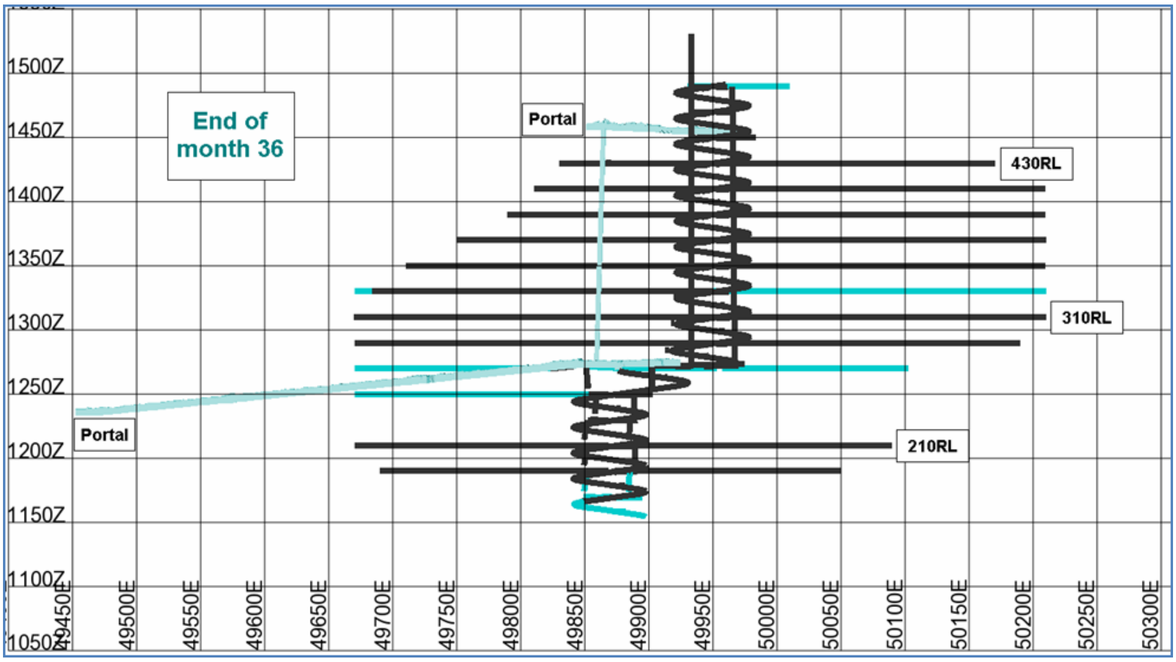


Figure 59. Development, end of Month 36.

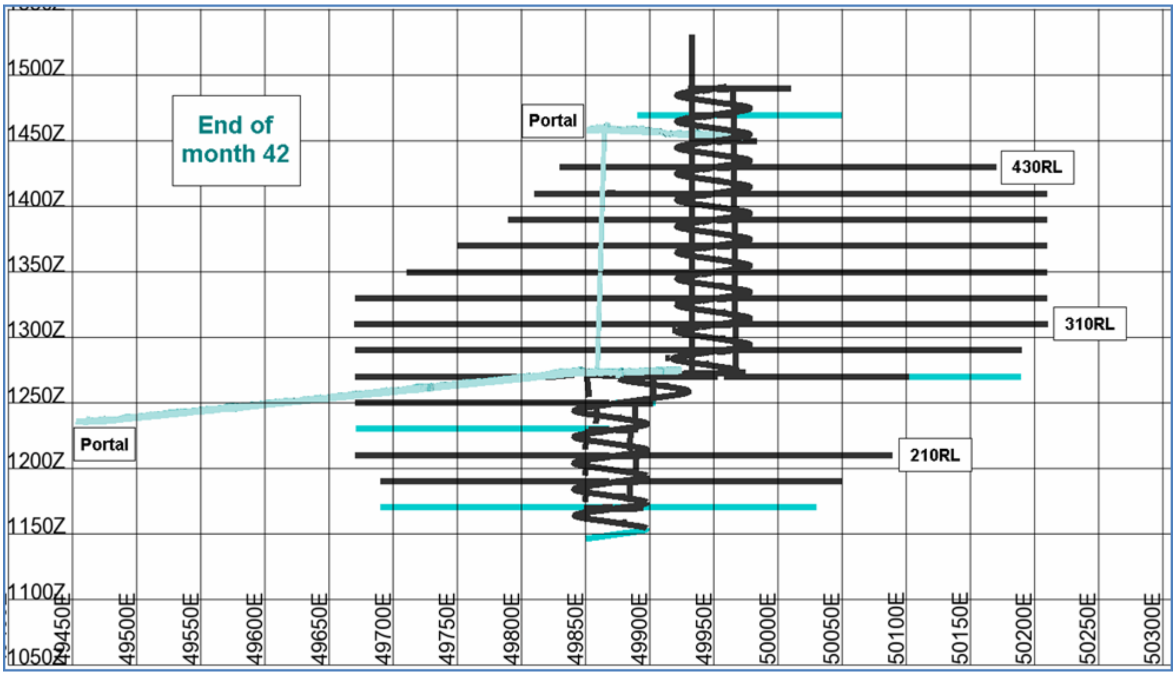


Figure 60. Development, end of Month 42

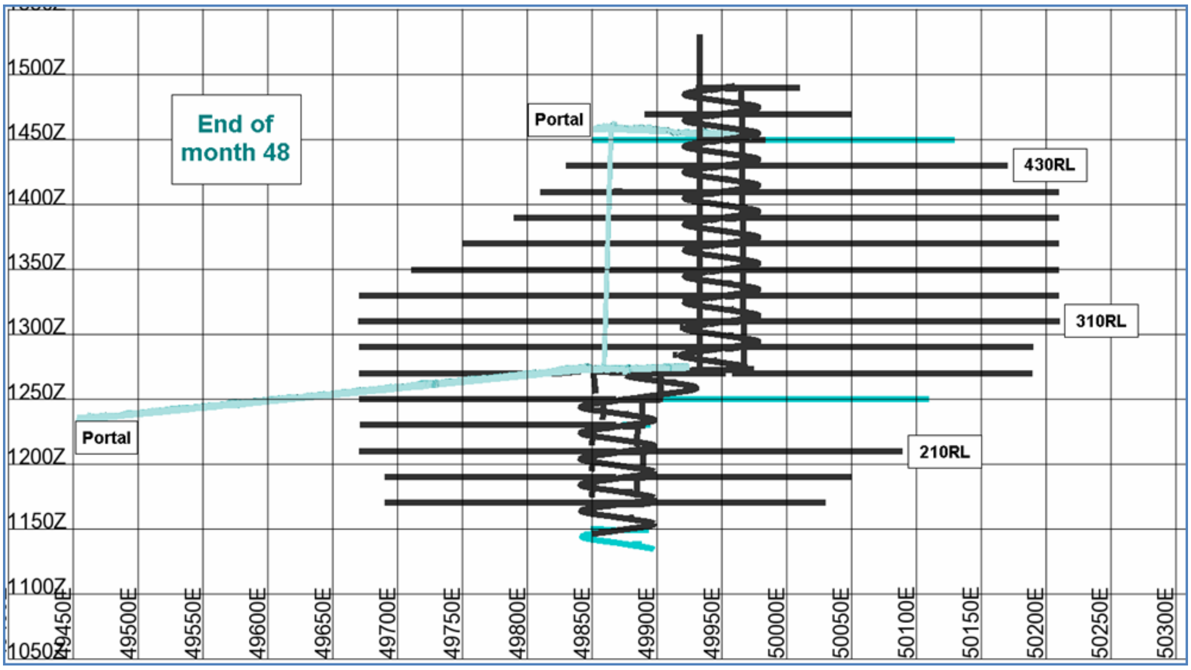


Figure 61. Development, end of Month 48

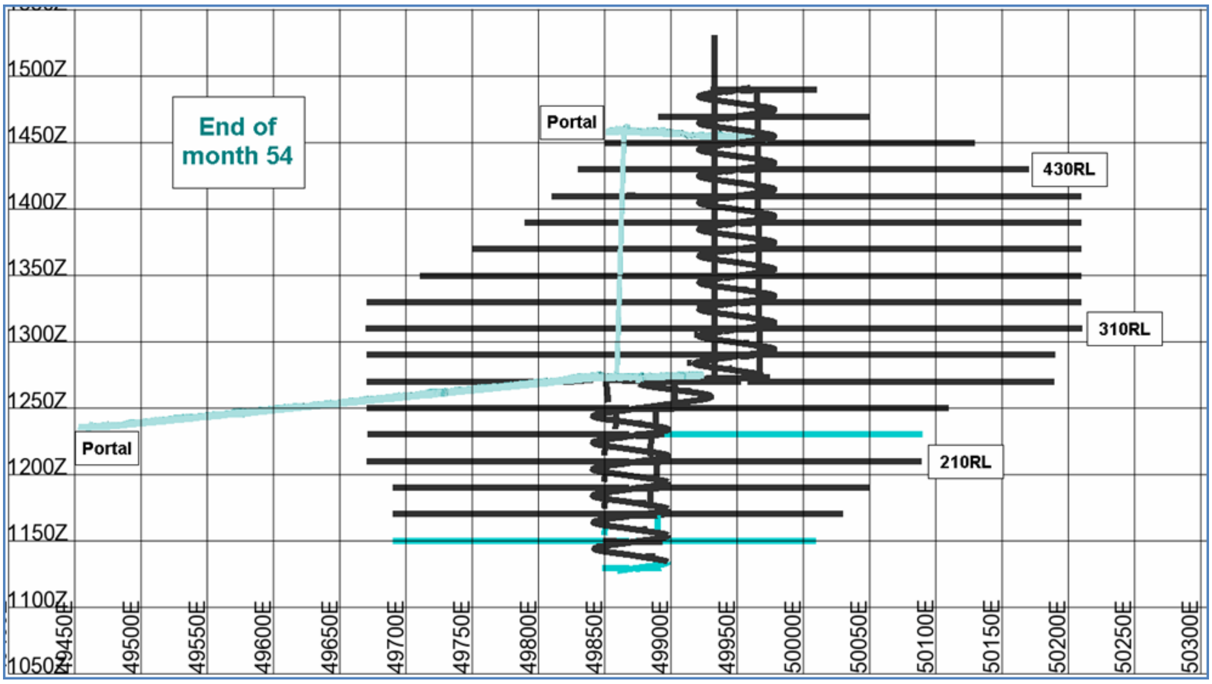


Figure 62. Development, end of Month 54.

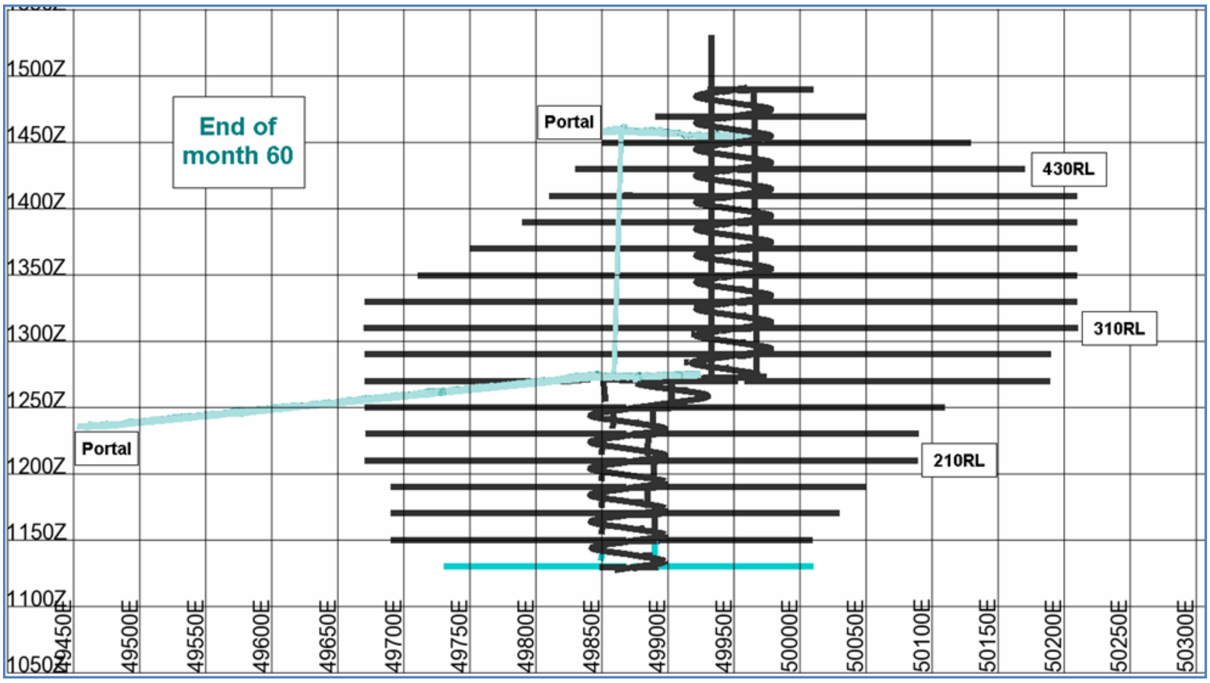


Figure 63. Development, end of Month 60.

Table 60. Schedule Quantities by Year

			Total	Yr1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Orebody Development	tonnes		**519,603**	82,658	141,814	131,976	90,738	72,418	0
	%	Ni	**2.03**	1.97	2.14	2.20	2.10	1.47	0.00
	%	Cu	**0.95**	1.06	1.00	0.99	0.95	0.65	0.00
	%	Co	**0.05**	0.05	0.05	0.05	0.05	0.04	0.00
Production	tonnes		**1,093,810**	41,140	199,812	224,975	253,589	293,239	81,055
	%	Ni	**2.30**	1.96	2.50	2.19	2.35	2.42	1.66
	%	Cu	**1.04**	0.95	1.12	1.04	1.03	1.06	0.79
	%	Co	**0.05**	0.04	0.06	0.05	0.05	0.06	0.03
Total Production	**tonnes**		**1,613,413**	**123,798**	**341,626**	**356,951**	**344,327**	**365,657**	**81,055**
	%	Ni	**2.21**	1.97	2.35	2.19	2.28	2.23	1.66
	%	Cu	**1.01**	1.02	1.07	1.02	1.01	0.97	0.79
	%	Co	**0.05**	0.05	0.06	0.05	0.05	0.05	0.03
	%	Ni Eq	**2.51**	0.00	0.00	0.00	0.00	0.00	0.00

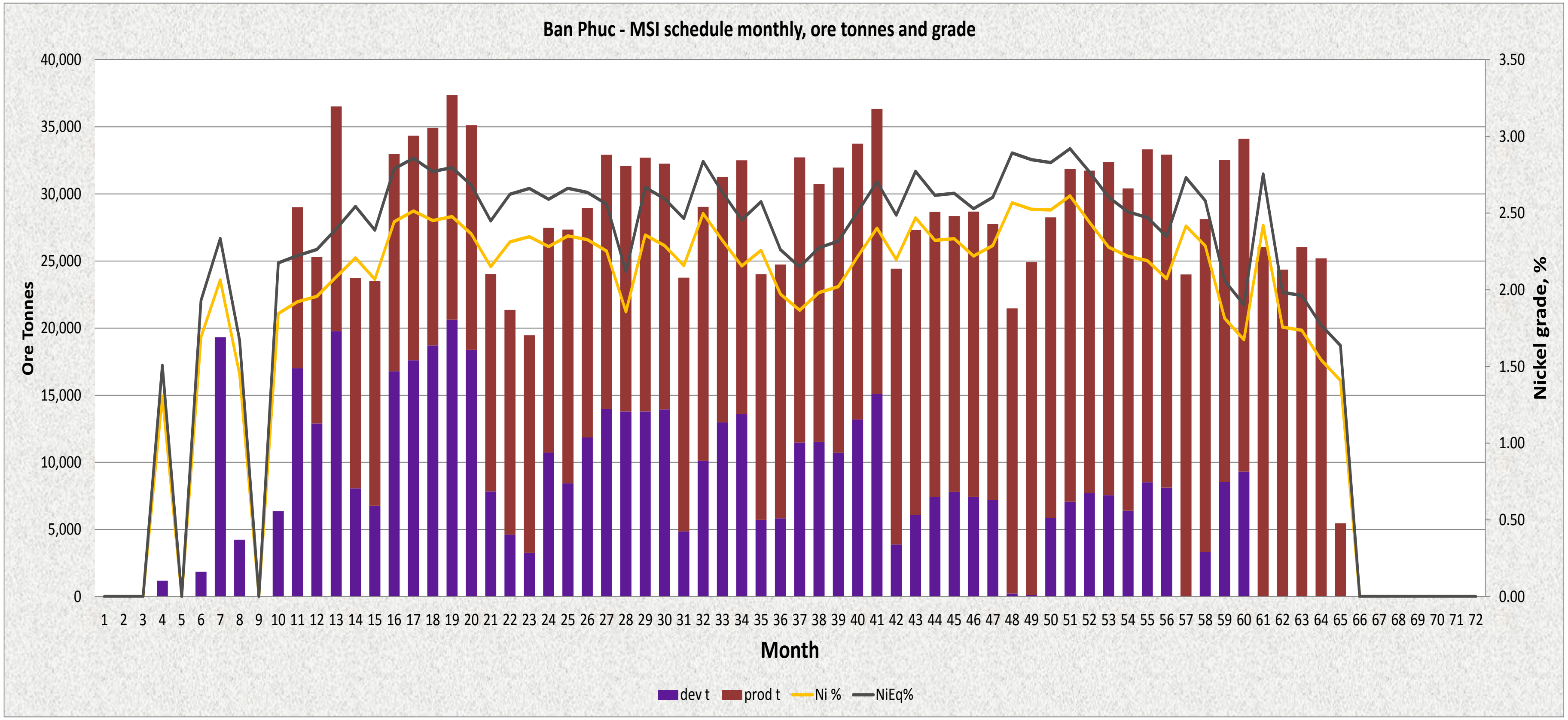


Figure 64. Monthly Ore tonnes and Nickel grade

Table 61. Mining fleet requirements.

			Yr1	Yr2	Yr3	Yr4	Yr5	Yr6
Unit	**Description**	**Max #**						
Development Drill	Small unit to work in 4m wide drives Eg Sandvik twin boom DD321. These units can also install ground support	2	2	2	1	1	1	
Production Drill	Small production rig capable of drilling 64 to 89mm holes. Also used for drain holes, cable holes, Ground support Eg Sandvik DL311 (formerly Tamrock Solo5). Dills 64 to 89mm holes	2	1	2	2	2	2	1
Orebody LHD	Small LHD to muck and tram in the orebody Eg Sandvik LH307 (formerly Toro 6). 6.5t capacity	2	1	2	2	2	2	1
Truck loading LHD	Slightly larger LHD to load from base or orepass into trucks. Eg Sandvik LH401 (formerly Toro 7). 10t capacity	1	1	1	1	1	1	1
Truck	For use in loading in declines and from orebody crosscuts. Eg. Atlas Copco MT2010 (20t)	2	2	2	2	2	2	1
Truck	For loading from orepass and other duties Eg Sandvik TH430 (30t)	2		2	2	2	2	1
Charge vehicle	For Drill and Blast duties Eg Cat It14G or similar	2	1	2	2	2	2	1
Service vehicle	To assist with services, ground support etc. Eg Cat It14G or similar	1	1	1	1	1	1	1
Light vehicle	Transport of workers, tradespeople and supervisors around the mine Eg Toyota Land Cruiser or similar	5						

17 Recovery Methods

17.1 Process Plant Description

17.1.1 Background

The process plant design has progressed through several iterations since the initial Feasibility Study conducted in 2005. It is important to understand these iterations, as substantiating documentation has moved from that of third party Feasibility Study and EPCM through to the current status of Owner Builder. These iterations have primarily focussed on increased process plant throughput and subsequent modification of equipment sizing and physical design, whilst the process flowsheet and metallurgical concepts have remained relatively consistent, with the exception of some minor modifications.

It is understood from interviews with site personnel and review of supporting documents that plant throughput design has evolved via the following chronology:

- Nov 2005 Feasibility Study – Ausenco - Plant throughput 200,000 tpa.
- 2007 Engagement of Metplant Engineering Services – Nominal throughput 250,000tpa; maximum throughput 300,000tpa.
- 2 November 2007 EPCM agreement signed by Metplant Engineering Services to build process plant.
- 12 May 2008 Change order to Metplant Engineering Services to increase plant capacity to 450,000tpa.
- 2 October 2008 Notification from Ban Phuc Nickel Mines to Metplant Engineering Services of Force Majeure, due to impact of Typhoon Hagupit, with subsequent suspension of all construction and related activities.
- It was understood that the project remained suspended for 12-24 months due to the Global Financial Crisis 2008 and subsequent lack of funding/support for re-commencement of the project.
- Sep 2010 Ban Phuc Nickel Mines personnel conducted interviews with 12 Engineering Groups, with a view to re-commencement of EPCM activities to complete the plant.
- Nov 2010 Two Australian Engineering companies, GR Engineering Services and Abesque Engineering, were invited to site to perform a project completion review for their Ban Phuc Nickel Mine.
- A decision was made by Ban Phuc Nickel Mine at some point after receiving quotations from GR Engineering Services and Abesque Engineering to discontinue the concept of a third-party EPCM approach, and adopt an Owner-Builder philosophy, utilising the 450,000tpa design information from Metplant Engineering Services.

- The design throughput of 450,000tpa is significantly greater than that required by the scheduled mine ore production to the plant. Target annual ore production increases to approximately 360,000tpa. It is thus clear that the processing plant has significant excess capacity.

17.1.2 General

The information produced below is generated from an assortment of working design documents from Metplant Engineers, understood to be generated in 2008 prior to the cessation of operations and EPCM activity. The previous 2007 Technical Report is based upon throughputs of 250,000tpa and 300,000tpa, with a reasonably detailed process description. The Metplant Process description for the upgraded plant throughput of 450,000tpa is not available. Hence, the information below has been developed from the 450,000tpa design criteria document provided by Ban Phuc Nickel Mines, and the assumption that the flowsheet remains unchanged.

It is understood that Metplant (now part of the Bateman Tenova group) is producing a summary design document for future reference.

The process plant is understood to be designed to process up to 450,000 tpa. The process plant will produce a bulk nickel/copper concentrate and will comprise a number of unit processes:

- Crushing;
- Crushed ore storage, reclaim and mill feed;
- Grinding;
- Flotation;
- Concentrate thickening, filtration, storage and load-out;
- Tailing thickening, pumping and return water;
- Reagent storage, mixing and distribution; and
- Utilities.

Process design criteria have been derived from metallurgical testwork and from these flowsheets and equipment lists have been prepared to develop the process plant design. The process plant location is located close to the mine portal. The main process plant is at one level and the concentrate shed situated 10 m below to accommodate the fall in the valley.

17.1.3 Crushing, Crushed Ore Storage, Reclaim and Mill Feed

Due to the ore’s potential to oxidize relatively rapidly, the storage capacity of the ROM ore stockpile will be kept to a minimum, preferably less than 5,000 t. The crushing circuit is situated next to the ROM ore stockpile. ROM ore will be reclaimed from the ROM ore stockpile. The crushing circuit has a design throughput rate of 79 tph, and a design availability of 65%.

The ROM bin will be fed via a duty front end loader through a 500 mm square aperture static grizzly into a 45 t live capacity ROM bin. The loader will be required to periodically remove oversize material retained by the static grizzly.

Grizzly undersize ore will be reclaimed from the ROM bin (Figure 66) by a vibrating feeder and will be directly fed into a single toggle jaw crusher. The vibrating feeder will be variable speed to control crushing circuit throughput. A single toggle crusher, selected to handle 500 mm maximum lump size and hard material, will crush the ROM ore prior to classification.

Primary crusher discharge (Figure 68) will be directed via conveyor to a vibrating double deck screen with 25 mm and 12 mm square aperture woven wire decks. Screen oversize will report via conveyor to a secondary cone crusher with the crusher product recycled to the screen. The mid-size will report to a tertiary cone crusher, with the product being recycled to the screen undersize which will discharge at a P80 product size of 9 mm and report via conveyors to the fine ore bin.

A fixed tramp metal magnet will be installed at the head of the screen feed conveyor to remove tramp iron as primary protection for the cone crushers. Secondary tramp metal protection will be provided by a metal detector on the screen oversize conveyor. This will be set up to mark the ore bed and stop the conveyor when alarmed.

Dust suppression water sprays will be provided at the ROM bin and at the crushers and vibrating screen; an insertable element filter will be provided on the fine ore bin to minimize dust emissions. The sprays will operate automatically when ore is dumped to the ROM bin and when the crushing circuit runs.

The fine ore storage building (Figure 67 and Figure 69) will provide 450 t live capacity.

All plate work within the crushing circuit will be lined with abrasion resistant liners at wear zones. The crushed ore will be reclaimed from the fine ore storage building by belt feeders onto the mill feed conveyor. In the case of a planned crusher shutdown, crushed ore can be stockpiled if necessary to provide several days feed to the mill. A belt weightometer will be installed on the mill feed conveyor to provide feed rate indication for control of the reclaim feeders and totalization of mill feed. Dust suppression sprays will be installed at the reclaim feeders and bin bypass diverter chute to minimize air borne dust.

Figure 66 ROM Bin

Figure 67 Crushing Circuit and Fine Ore Storage

Figure 68 Locally Manufactured Primary Crusher

Figure 69 Fine Ore Storage

17.1.4 Grinding

The reclaimed crushed ore will be fed at a controlled rate to the ball mill. The dimensions of the procured second-hand ball mill are 3.6 m diameter by 6.2 m flange to flange. If required, the mill can be shortened and refurbished to match the overall capacity of the Ban Phuc process plant.

Discharge from the ball mill (Figure 70 and Figure 71) will gravitate through a rubber lined trommel and into the mill discharge hopper. Scats from trommel oversize will be collected in a bunker for recycle or disposal. The ball mill will be rubber lined to avoid the need for specialized liner handling equipment.

The ball mill discharge slurry will be pumped to a cyclone cluster operating in closed circuit configuration with the ball mill, with as design cut-size of 90 µm. Cyclone underflow will report back to ball mill feed. Cyclone overflow will gravitate onto a trash screen. A cluster of three cyclones will be installed, two of which will be in operation with one on standby. A manual valve will be provided to isolate each cyclone. Ore chutes will be lined with abrasion resistant liners at wear zones. A vertical spindle sump pump will be provided in the grinding area to facilitate clean up.

A flash flotation cell, installed beneath the cyclone cluster, will receive cyclone underflow feed, with flotation concentrate reporting to final concentrate, and flash flotation tailing gravitating back to the ball mill feed box.

Figure 70 Ball Mill Foundations

Figure 71 Milling Section Construction

17.2 Flotation

17.2.1 Feed Preparation

Trash screen underflow will gravitate to an agitated rougher flotation conditioning tank via a full-stream sampler.

17.2.2 Rougher Flotation

The rougher flotation conditioning tank will discharge to the rougher flotation cells connected in series. A bank of five 16 m3 forced air tank cells have been selected providing the required residence time for roughing duties. The cells will be installed in a 3 + 2 configuration.

Concentrate from the rougher cells will be pumped to the cleaner feed. An additional pump will be installed with an option to deliver the product from the last two cells to the cleaner scavenger rather than the cleaner feed. Rougher tailings, combined with tailings from the cleaner scavenger bank, will flow to the flotation tailings hopper. An automatic sampler will be located on this stream. Flotation tailings will be pumped to the tailings thickener. Duty and standby pumps will be provided for this duty. A vertical spindle sump pump will be provided in the rougher and scavenger flotation area to facilitate clean up.

17.2.3 Cleaner Flotation

Rougher concentrate will report to the cleaner cells, which are connected in series. A bank of five 8 m3 forced air flotation cells (Figure 72 and Figure 73) will be installed in a 3 +2 configuration, to provide the necessary residence time for process requirements. Concentrate from the cleaner cells will report to the final concentrate pump and will then be pumped to the concentrate thickener. An automatic sampler will be installed on this stream. Tailings from the cleaner cells will flow by gravity to the cleaner scavenger cells.

The pumping system will be designed so that cleaner scavenger concentrate can be directed to final concentrate if the grade is suitable.

Alternatively the concentrate from the cleaner scavenger cells will be pumped back to the cleaner cells. Tailings from the cleaner scavenger will gravitate to the combined final tailings hopper. A vertical spindle sump pump will be provided in the cleaner flotation area to facilitate clean up.

In addition, the size and number of flotation cells for roughing and cleaning duty has essentially remained unchanged despite the throughput increase from 300,000tpa to 450,000tpa. The key mitigating factor is the installation of a flash flotation cell (Figure 76).

Figure 72 Flotation Section under Construction

Figure 73 Flotation Cells

17.2.4 On-Stream Analysis (OSA) and Sampling

Flotation feed, combined rougher and scavenger concentrate, scavenger tailing, cleaner scavenger tailing and cleaner concentrate streams will be sampled for metallurgical control by a combination of pump discharge pressure samplers, launder fixed cutter samplers and sample pumps and directed to the OSA unit. This selection of samples will allow effective metallurgical control of the rougher circuit and cleaner circuit.

The OSA system will be located at a position which will permit maximum gravity flow of sample feed and reject slurries to minimize requirements for sample feed and return pumps.

17.2.5 Concentrate Dewatering

Final concentrate will be pumped to a 9 m diameter high-rate thickener via a feed de-aeration box. Thickener overflow will gravitate to the process water tank and thickener underflow will be pumped to the concentrate stock tank by variable speed centrifugal slurry pumps.

Sufficient ground clearance has been provided under the thickener cone for maintenance and general clean up. The agitated concentrate stock tank has approximately 17 hours total surge capacity. The agitator will be selected with the lower impeller placed as deep in the tank as practicable to allow the tank to be drawn down as close to empty as possible by the filter feed pump at the tank base. This will maximize tank live volume.

Thickened concentrate slurry will be pumped to the filter press during the fill cycle using a variable speed centrifugal slurry pump. At other times, concentrate will be recycled back to the concentrate stock tank. The filter press will reduce the moisture content of the concentrate prior to transport. Filter cake will discharge through the floor onto a conveyor belt which will deliver it to the concentrate shed.

Filtrate from the filter press will be returned to the concentrate thickener feed de-aeration box. The filter press will be housed in a separate building, Compressed air for the filter press will be provided from a dedicated filter air compressor and receiver with sufficient capacity to cater for surges in compressed air demand during the filter cycle (Figure 74).

The concentrate shed (Figure 75) has been constructed at a lower level so the conveyor will feed into the top of the shed. Concentrate will be reclaimed from under the conveyor and loaded into bulka-bags. These will be filled via a hopper which will be fed by the loader. A weighing system will be incorporated into the bagging facility. An area sufficient for about seven days storage of bulk concentrate at the maximum production rate will be provided. Space will be allocated to stockpile concentrates of different grades. A floor sump and vertical sump pump will be provided to facilitate clean up.

Figure 74 Thickeners, Reagent section and Filter Feed Tank

Figure 75 Concentrate Filter building and Concentrate Storage Building at lower level

The majority of remaining process plant capital equipment remains in storage on site, primarily within or adjacent to the Concentrate Storage Shed, as show in Figure 77 to Figure 81.

Figure 76 Flash Flotation Cell and Cyclone Nest

Figure 77 Ball Mill Girth Gear

Figure 78 First Fill Ball Mill Liners

Figure 79 Ball Mill Bearings and Spares

Figure 80 New Capital Equipment in storage

Figure 81 New Capital Equipment awaiting installation

17.2.6 Tailings Dewatering

Flotation tailings will be pumped to a 9 m diameter high-rate thickener for dewatering. Thickener underflow will be pumped to the tailings storage facility (TSF) and thickener overflow will gravitate to the process water tank. The TSF has been designed by Knight Piésold. Sufficient ground clearance has been provided under the thickener cone for maintenance and general clean up.

Flocculant will be dosed to the tailings thickener feed box to aid in settling and to provide the necessary clarity in thickener overflow. Supernatant water from the TSF will be returned to the process water tank using skid mounted pumps with floating suctions.

17.3 Reagents

17.3.1 pH Modifier– Sodium Carbonate (Soda Ash)

Sodium carbonate will be delivered to site as a solid in bulk bags. These will be lifted into a sealed bag splitter enclosure discharging into a mixing tank. Mixed solution will be transferred to a storage tank and reticulated in a ring main system to various usage points.

17.3.2 Collector - Sodium Ethyl Xanthate (SEX)

SEX will be supplied as a solid in 120 kg drums. Drums will be lifted into a sealed enclosure consisting of a roller conveyor and tipper. SEX will be mixed in an agitated and ventilated tank, and then transferred to a storage tank. SEX solution will be reticulated in a ring main system to various usage points.

17.3.3 Frother – InterFroth

IF50 frother will be supplied as a liquid in 1,300 kg containers and coupled directly to circulating pumps as each drum provides approximately 14 days usage. The IF50 frother will be reticulated in a ring main system to various usage points.

17.3.4 Sulphide Gangue Depressant - Sodium Metabisulphite (SMBS)

SMBS will be supplied as a solid in 25 kg bags. The bags will be lifted into a sealed bag splitter enclosure discharging into a mixing tank. Mixed solution will be transferred to a storage tank with capacity for approximately 48 hours consumption and reticulated in a ring main system to various usage points.

17.3.5 Non-Sulphide Gangue Depressant - Carboxy Methyl Cellulose (CMC)

CMC will be supplied as a solid in 25 kg bags. These will be lifted into a bag splitter enclosure discharging into a proprietary skid-mounted venturi jet mixing system. Mixed solution will be transferred to a storage tank with capacity for approximately 24 hours consumption and reticulated in a ring main system to various usage points.

17.3.6 Flocculant

Flocculant will be supplied as a solid in 25 kg bags. These will be lifted into a bag splitter enclosure discharging into a proprietary skid-mounted venturi jet mixing system. Mixed solution will be transferred to a storage tank with capacity for approximately 24 hours consumption. Flocculant will be dosed to the tailing thickener by variable speed helical rotor pumps. Table 62 provides reagent addition details.

Table 62 Reagents

Reagent Addition (g/t)						
Addition point	**Sodium Carbonate**	**Sodium Metabisulphite**	**Sodium Ethyl Xanthate**	**CMC**	**Frother IF50**	**Flocculant**
Grind	1,000	400				
Conditioner	500			200		
Rougher			55		40	
Cleaner	100	40	5	20	20	
Total	1,600	440	60	220	60	
Thickeners						35

17.4 Process Plant Utilities

17.4.1 Raw and Fire Water Distribution

Raw water will be sourced from bores and stored in a combined raw and fire water tank at the plant site. It will be used for purposes where process water is not suitable but potable water is not necessary e.g. reagent mixing, cooling, gland water and fire water and for feed to potable water treatment and for make-up to process water.

Operating and standby pumps will deliver raw water to distribution mains. The raw water outlets are part way up the tank so that there will always be a minimum reserve for fire water. Fire water will be delivered to a ring main by an electric driven primary pump with a diesel standby. The ring main will be kept pressurized by an electric jockey pump.

17.4.2 Potable Water Treatment and Distribution

Potable water for drinking will be commercialized bottled water. The local bore water can be used for other potable water usages such as showers as the only limitation on its potability is the nitrate content.

17.4.3 Process Water Supply and Distribution

Internal recycling of tailings thickener and concentrate thickener overflows and concentrate filtrate will be the main source of process water. With a positive water balance of catchment over evaporation for the site, and for the TSF in particular, most process water will be sourced from tailings reclaim.

For a significant proportion of the year it is expected that there will be a surplus of water from the TSF and no make-up of process water with raw water will be required. However appropriate provision has been made for dry spells and for plant start-ups.

17.4.4 Compressed Air

High pressure compressed air will be supplied by an oil-free compressor for general plant air with back up from a mobile compressor shared with the underground mine delivering through an air filter to remove oil. Instrument air will be obtained by refrigerated drying of plant air. Compressed air in the process plant will be generated oil-free to minimize contamination of the flotation process and to minimize the load on the refrigerated dryers for instrument air. Local oilers will be provided for air tools.

One duty and one standby blower will provide low pressure air to the flotation cells. The blowers will be variable speed to cater for fluctuations in flotation air demand. Each flotation bank will be equipped with a flow meter and a control valve for the controlled addition of air. A dedicated compressor is included in the concentrate pressure filter package.

17.4.5 Power Distribution

Power is delivered to the site from 6.6 kV incomers from the power authority's high voltage ("HV") switchroom. The site main substation includes HV distribution to the other site substations and to step down transformers for supply of lower voltage HV power (1 kV) and low voltage ("LV") three phase power (415 V). Other substations are located at the underground haulage and access portals, and camp. Each substation incorporates breakers for HV and LV power, step-down transformer(s), Motor Control Centres ("MCCs") for three phase alternating current ("AC") motors and distribution panels for single phase AC consumers and for control and instrumentation supplies.

For transmission within the plant site and other on-site infrastructure, insulated power cables are run preferentially in cable tray on racks or below ground. Overhead conductors transmit power to consumers too distant from substations for the use of insulated cable, i.e. the mine portals and staged tailings pumps. Other minor consumers remote from substations are powered by local diesel generator sets.

17.5 Process Control

The general control system for the process plant is one with a moderate level of instrumentation and automation and takes into account the following considerations:

- The limited availability of instrument technicians experienced in mineral processing plants would make maintenance of a heavily instrumented and automated plant difficult.

- The remote location and the limited scale of Vietnam's mining industry suggest that spare parts and technical support from suppliers and contractors will be limited.
- The small scale and limited life of the Project makes expenditure of additional capital to achieve operating efficiencies difficult to justify except for significant improvements. The relatively low remuneration of Vietnamese national personnel limits the incentive for reducing the number of personnel.

The following factors weigh in favour of a moderate level of instrumentation and automation:

- The limited availability of operations technicians experienced in mineral processing plants would make efficient operation and optimal performance difficult to achieve without some level of automatic control.
- There is a minimum level of instrumentation and automation required for personnel safety and protection of equipment, consistent with BPNM's aim of performing to sound international mining industry standards. The control system developed is typical of modern mineral processing operations of this scale in similar locations.

The plant control system will be a PLC based SCADA with PC-based operator control stations. The SCADA system will be configured to provide outputs to the graphical user interface for alarms, control of the function of selected process equipment and provide logging and trending facilities to assist in analysis. Field instrumentation will provide inputs to PLCs on process and equipment conditions.

The PLCs will also collect status information on all process drives as well as providing drive control and process interlocking.

The operator control station will be installed in the control room and provide the following functions:

- Graphic (mimic) displays of all plant areas, showing equipment status (ready, not ready or running) and values for critical process variables.
- Alarm display and logs, showing the alarm tag number, title, date and time.
- Trend displays with flexible time and process variable axes for any analogue process variable.
- Control loop displays showing controller settings and trending of process variable, set point and output.

17.6 General Drive Control

Most equipment will be started via the SCADA wherever possible. Equipment drives will have two modes of operation:

- Auto, and
- Maintenance (manual).

In Auto mode the drive will be started and stopped from the PLC/SCADA under the control of the PLC program, either in an automated sequence or directly by the operator in the control room. Equipment will only be able to be started and stopped in particular sequences to ensure personnel safety is maintained and equipment is protected from poor operating conditions such as overloading.

In Maintenance mode, the operator will start and stop the drive from a field stop/start push button station located adjacent to the drive. The field start push button will only function in Maintenance mode, but the stop button will always be functional.

Maintenance mode will also allow the operator to start equipment out of sequence and without process interlocks. Safety interlocks such as conveyor pull wire switches, which must remain operational at all times, will be hardwired. All other interlocks will be implemented in the PLC.

17.7 Common Control Functions

A number of common control functions will be provided throughout the plant. These are described as follows and are relevant to each area unless noted otherwise.

17.7.1 Conveyor Drives

In addition to the general drive configuration, conveyor starter modules will be provided with a start delay timer to energize an audible alarm for 10 seconds prior to the conveyor motor being energized. In addition the following field devices will be provided:

- Pull-wire switches on the sides of the conveyor where accessible. These switches will be hardwired to the conveyor stop circuit.
- Limit switches to determine belt drift, located at the head and tail ends.
- Speed sensors to detect low speed or belt slip.

17.7.2 Chutes

Where applicable, chutes will be fitted with a level switch to alarm on at high level and stop equipment feeding the chute when there is a blockage (i.e. blocked chute switch).

17.7.3 Sump Pumps

In addition to the general drive configuration, sump pumps will be provided with a de-contactor plug and socket to enable quick disconnection in the field. A power junction box will be provided adjacent to each drive which will house the start push button, lock-off stop and de-contactor socket.

Operation of the sump pumps will be automatic. Each sump will be fitted with a float switch which activates at a high level in the sump. The high level signal will start the pump which will run for a present time. If the switch indicates the sump is still at a high level after the timer has expired, an alarm will be registered at the SCADA.

17.7.4 Mass Flow Measurement

For both control and tonnage accounting purposes at various points in the process mass flow measurement will be provided.

For dry material on conveyors this will be achieved by belt weightometers. These have a local display of instantaneous belt loading and integrated wet tonnage flowrate, as well as totalized tonnes. A moisture content factor used for mass balancing would be entered in the SCADA and used for on-line dry tonnage calculations. The dry tonnage flow rate and totalized dry tonnes will be calculated by the PLC and displayed on the SCADA system in addition to the wet tonnage flow rate and totalized wet tonnes.

For slurry flows, magnetic flow meters and a nuclear density meters will be installed on the relevant pipeline. Instantaneous slurry density and volumetric flow will be displayed locally and on the SCADA system. The dry tonnage flow rate and totalized dry tonnes will be calculated by the PLC and displayed on the SCADA system.

17.7.5 Crushing and Fine Ore Storage

The main area of automation will be personnel safety and equipment protection. Traffic signals will be located on the ROM pad to advise the front end loader operators whether there is sufficient room in the ROM bin for dumping. The traffic light will be automatically actuated, based on the level indication monitoring of the bin, or manually actuated by the operator.

The crushing plant is locally manufactured and has limited automation. The primary crusher vibrating feeder will be rate controlled from the control room. Operating parameters for the secondary and tertiary crushers will be displayed in the control room but settings will be manual.

The level in the fine ore storage building is measured by ultrasonic level sensor. If the level rises above a high set point, then the primary crusher feeder is stopped until the level falls below a middle set point. If the level rises above a high-high set point, then the fine ore storage feed conveyor is stopped.

In addition to the general drive configuration, the primary and secondary crusher starter modules will include current transducers for monitoring of motor currents locally in the MCC and via the plant SCADA system.

17.7.6 Fine Ore Reclaim and Grinding

After personnel safety and equipment protection, the main control driver in the grinding circuit is to maintain steady throughput and consistent flotation feed pulp density and particle size.

The ball mill will be driven by a wound rotor induction motor. A liquid resistance starter is part of the unit being purchased second hand which will enable soft starting of the mill.

The mill starter will include power draw measurement and a kilowatt-hour meter. The power will be displayed and trended on the SCADA system.

The ball mill will be supplied with a mill control panel which will house all the starters and controls for the mill auxiliary drives. The panel will include a PLC for control and monitoring that will be connected to the overall plant SCADA network.

The ball mill will be will be started from a local control panel. A start siren provided on the mill control panel will sound on start-up of the mill. Individual starting of mill auxiliary drives will be from the mill control panel only. As well as regulating the starting and stopping of equipment, the control system will provide automatic control loops in the grinding circuit for the following:

- Ball mill feed rate control – The fine ore reclaim (ball mill feed) rate will be controlled to a tonnage set-point using the variable speed drive of the fine ore feeder (or the fine ore bypass feeder as required).
- Ball mill feed water addition – Water addition will be measured by a flow meter and regulated by a flow control valve. Mill feed water will be controlled automatically in a set ratio to the ball mill feed rate to control the pulp density of the mill charge.
- Ball mill discharge water addition – Water addition to the cyclone feed pump hopper will be regulated to maintain a set ratio of overall grinding water addition to the ball mill feed rate to control cyclone overflow pulp density. This control loop will be set up with a slow response time to minimize unstable conditions introduced due to short term feed stoppages.
- Cyclone feed pump speed control – The cyclone feed hopper level will be measured and automatically controlled by the variable speed mill cyclone feed pump. To minimize the variation in flotation feed flow rate, the level in the cyclone feed hopper will be allowed to vary between high and low set points. If the level rises above or falls below the respective set point, then the pump speed will be increased or reduced accordingly. If the level rises above a high-high set point or falls below a low-low set point, then there will be an alarm to alert the operator.

The ball mill power draw will be continually monitored and controlled by the addition of grinding media. Ball addition will be conducted manually and is expected to be performed on a daily basis.

17.7.7 Flotation

All electric drives in the flotation circuit can be started from the main control room. In addition to the general drive configuration, the flotation cell agitator starters will include current transducers for monitoring of motor currents locally in the MCC and via the plant SCADA system. Drive reversing switches will also be provided on the MCCs for each of the flotation cell agitator drives.

Apart from regulating starting and stopping of equipment, the control system will also provide automatic control loops for the following:

- Flotation stage air flow control – The rougher, scavenger, cleaner and scavenger cleaner flotation banks will be equipped with dedicated low pressure air headers. The operator will be able to set an air flow to each of these flotation stages and an automatic controller will ensure that the operator required air flow is maintained by regulating the position of a flow control valve.

- Flotation cell froth level control – Each flotation bank slurry/froth interface level will be controlled by regulating the position of the pneumatically actuated float cell discharge pinch valve. The operator will set a froth depth level and the controller will monitor the interface level from the input provided and regulate the position of the discharge pinch valve to maintain the required level.
- Pump control – The concentrate pumps in the flotation circuit will be vertical spindle pumps located in concrete sumps. The pumps will be set at fixed speed as there will be internal recycling if the pump runs dry. A high level alarm will activate when the sumps are full. Tails slurry transfer pumps in the flotation circuit will be speed controlled to maintain an operator desired pump hopper level. The operator will enter a pump hopper level into the specific controller and a level controller will regulate the pump drive variable voltage, variable frequency ("VVVF") to maintain the set-point.
- Reagent addition control – The pH will be constantly measured at a number of points within the flotation circuit and automatically controlled by the addition of soda ash from a ring main. Collector, frother and sodium metabisulphite additions will be manually controlled by the plant operators using dosing pumps.
- The flotation performance will be monitored using an OSA system performing on-line analysis for nickel, copper, iron and slurry density. OSA measurements will be transmitted to the SCADA system and will allow calculation of metal recovery, warnings and trend displays.
- The flotation air pressure will be continually monitored and displayed on the SCADA system. Alarms will be configured to alert the operators of low and high pressure events, although the header pressure will be manually controlled by the adjustment of a blow-off valve.

17.7.8 Thickeners

Bed pressure sensors will be installed on each thickener. A nuclear density gauges will be installed on the thickener underflow pump delivery pipeline. The bed pressure sensor or the nuclear density meter will control the thickener underflow pump variable speed, i.e. slowing the pump if the bed pressure or underflow slurry density drops and speeding up the pump if the bed pressure or density rises. A very high bed pressure will override other control strategies taking the pump to full speed until the condition is cleared to avoid bogging the thickener.

A magnetic flow meter will be installed on the delivery line from the underflow pump to the downstream process. The volumetric flow and nuclear slurry density measurement will be integrated in the PLC to provide indication and totalisation of the dry tonnage flow rate to the downstream process.

The vendor provided thickener control panel will control the rake lifting and lowering to ensure that the rakes are protected against overloading. Appropriate alarms will be configured to alert operators of high rake torque and high rake height conditions.

The thickeners will be equipped with interface detectors that warn the operator via the SCADA if the interface is rising so that corrective action can be taken before a sliming condition occurs. Thickener rake torque measurement will be provided and displayed and logged for trending on the SCADA system. If an increase in solids loading raises the torque, then high and high-high alarms will be activated at the SCADA. The high-high alarm shuts down the plant to avoid bogging the thickener. The interface level will be used to control the rate of flocculant addition.

17.7.9 Concentrate Filtration

The concentrate filter will be supplied with a dedicated vendor control panel and interfacing system for parameter setting within a dedicated PLC. This PLC will control the speed of the filter feed pump and sequencing of the filter operating cycles. There will be a general fault input from this panel to the plant PLC to alert the control room operator of fault conditions.

A siren and flashing light will be installed to alert nearby personnel of the likelihood of moving machinery and discharging concentrate.

17.8 Reagent Make-up and Distribution

17.8.1 pH Modifier

Batches of sodium carbonate (soda ash) solution will be mixed as required. A level probe on the soda ash storage tank will provide a low level alarm to alert the control room operator when another batch is required. Soda ash solution will be pumped from the storage tank through a ring main and dosed to the flotation circuit through solenoid operated valves. pH control loops will control soda ash addition rates (solenoid timers) at several points in the circuit. A low flow switch will be installed in the ring main to provide a low flow alarm.

17.8.2 Collector

Sodium ethyl xanthate will be supplied in pelletized form and batches of solution will be mixed as required. A level probe on the SEX storage tank will enable the level in the tank to be continually monitored. A low level alarm will be configured to alert the control room operator when another batch is required. SEX solution will be pumped from the storage tank through a ring main. Dosing points at a number of locations around the flotation circuit will be controlled using dosing pumps.

17.8.3 Frother

InterFroth IF50 will be pumped directly from the container to a ring main. A level probe will provide continuous monitoring of the level in the container and a low level alarm will alert the operators to top up the container when required. Dosing rates at a number of locations around the flotation circuit will be controlled using variable speed dosing pumps.

17.8.4 Flocculant

The flocculant mixing system will be supplied complete as a vendor package with a dedicated control system. The vendor supplied control system will manage and monitor mixing of flocculant. However flocculant transfer to the flocculant storage tank will be based on level indication in the storage tank. A general alarm will be monitored by the process plant PLC. Helical rotor pumps will provide flocculant to the tailings thickener and the rate will be controlled by a variable speed drive on the pump.

17.8.5 Non-Sulphide Gangue Depressant

The CMC mixing system will be supplied complete as a vendor package with a dedicated control system. The vendor supplied control system will manage and monitor mixing of CMC. However, transfer to the CMC storage tank will be based on level indication in the tank. A level transmitter will provide status indication to the plant PLC.

A general alarm will be monitored by the process plant PLC. CMC will be circulated through a ring main and dosed to the flotation circuit using manual valves. A low flow switch will be installed in the ring main to provide a low flow alarm.

17.8.6 Sulphide Gangue Depressant

Sodium meta-bisulphite (“SMBS”) will be supplied as a powder and batches of solution will be mixed as required. A level probe on the storage tank will provide a low level alarm to alert the control room operator when another batch is required. SMBS solution will be pumped from the storage tank through a ring main. Dosing points at a number of locations around the flotation circuit will be manually controlled using needle valves. A low-flow switch will be installed in the ring main to provide a low-flow alarm.

17.8.7 Samplers

Flotation feed, combined rougher and scavenger concentrate, scavenger tailing, cleaner scavenger tailing and cleaner concentrate streams will be sampled for metallurgical control by a combination of pump discharge pressure samplers, launder fixed cutter samplers and sample pumps, and directed to the OSA unit. Samplers or pumps will be activated as necessary through the control system.

17.8.8 Fire Water Pumps

The fire water pumping system will consist of an electric fire water pump, standby diesel fire water pump and small jockey pump. The electric jockey pump will maintain pressure in the firewater distribution piping and the main electric pump will start if the line pressure cannot be maintained. The diesel drive pump will start if the electric pump fails for any reason and will need to be manually stopped at the local control panel once it has started.

17.8.9 Potable Water

Potable water will be transferred to a storage tank from where distribution pumps will supply water to the process plant, and safety shower header tank. The emergency safety showers and eyewash units will be fitted with flow switches that will provide an alarm to the control room when a particular safety shower has been actuated.

17.8.10 Concentrate Production

Details the estimated annual concentrate production are outlined in Table 63.

Table 63 Project Concentrate Production Schedule

Year	Concentrate (DMT)	Concentrate (WMT)	Contained Nickel (t)	Contained Copper (t)	Contained Cobalt (t)
2013	30 852	33 783	2 931	1 712	56
2014	75 444	82 611	7 167	3 565	148
2015	68 582	75 098	6 515	3 465	125
2016	72 966	79 898	6 932	3 287	134
2017	67 033	73 401	6 368	3 190	122
2018	4 166	4 562	396	234	6
Total	319 043	349 353	30 309	15 453	592

The nominal concentrate nickel grade is approximately 9.5% Ni, with a high copper tenor (approximately 2:1 nickel:copper) and approximately 0.2% cobalt with no significant precious metal or platinum group metal values (see Table 64). The concentrate will have low levels of deleterious elements and, in particular low MgO, so are not expected to suffer any penalties.

17.8.11 Concentrate Quality

The terms being discussed with the smelter refer to a typical assay of the nickel concentrates produced at Ban Phuc. The concentrate analyses of the test concentrate is as set out in Table 64.

Table 64 Typical Nickel Concentrate Analysis

Element	Unit	Range
Ni	%	9.0 - 10.0
Cu	%	3.5 - 6.0
Co	%	0.28 - 0.32
Pt	ppm	0.1 - 0.5
Pd	ppm	0.1 - 0.3
Au	ppm	0.12 - 0.24
Ag	ppm	5 - 8
Fe	%	38.0 - 47.0
S	%	28.5 - 36.5
MgO	%	0.17 - 2.54
A12O3	%	0.3 - 1.7
SiO2	%	2.2 - 8.5
CaO	%	0.4 - 1.6
As	ppm	<10
Bi	ppm	<10
Cd	ppm	100 - 130
Cl	ppm	100 - 300
Cr	ppm	270 - 380
F	ppm	100 - 200
Hg	ppm	30 - 40
Mn	ppm	70 - 250
Mo	ppm	<5
Pb	ppm	<5 - 74
Sb	ppm	<5
Se	ppm	10 - 40
Sn	ppm	<10
Te	ppm	4.4 - 8.0
Zn	ppm	180 - 350

Notes: Moisture 7-8% Size: 80% passing 78 microns

18 Project Infrastructure

18.1 On Site Infrastructure

18.1.1 General

The process plant site is on a relatively gently sloped area of West Ban Phuc Valley. A small ROM ore pad is to be developed at the level of the underground mine haulage portal and the rest of the site is to be developed by cut and fill. Cut off drains are required around the site to divert the valley catchment run-off to downstream of the plant site pads:

- ROM ore stockpile and process plant at 220 RL;
- Concentrate shed at 210 RL;
- Workshop and warehouse at 200 RL; and
- Administration building at 200 RL.

Significant retaining structures have been installed on some of the cut slopes associated with the plant site. Some slopes are suitable for gabion walls but others have been secured by soil-nailing and mesh or shotcrete or combinations of these approaches. Additional slope control is recommended.

18.1.2 Tailings disposal and plant site runoff dam

The Project will produce approximately 280,000 tpa of tailings for just over five years. The tailings storage facility (TSF) has been designed for 1 Mt of tailings and is capable of being expanded to up to a maximum of 3 Mt.

The tailings may be acid generating so sub-aqueous tailings deposition techniques will be used and the tailings beach will be permanently covered by a minimum of 4 m of water to reduce oxidation. The TSF is located in Suoi Dam (Dam Creek) Valley approximately 3.5 km by road or 1 km in a straight line northeast of the West Ban Phuc process plant.

A plant site runoff dam (PSRD) will be constructed in the downstream end of West Ban Phuc Valley to control runoff and sediment from the mine and process plant site area. The general site layout is shown in Figure 5.1

The design objectives for waste and water management for the TSF and PSRD are:

- Permanent and secure containment of all solid waste materials
- Provide a safe method of disposal for potentially acid generating tailings using sub-aqueous deposition
- Removal and reuse of free water
- Minimization of seepage

- Provide a permanent overflow spillway with a capacity for a 1 in 500 year intensity storm event
- Provide a water supply for a 1 in 100 year dry condition
- Rapid and effective rehabilitation
- Ease of operation
- Sediment control for runoff from the mine and process plant

The design criteria for the TSF and PSRD details are:

- Storage of at least 1 Mt. Maximum requirement is to be confirmed but may be up to 3 Mt
- Embankment stability for an operating basis earthquake (OBE) with acceleration coefficient of 0.16g and peak horizontal ground acceleration of 0.28g for a maximum credible earthquake (MCE).
- Estimated mine life of five to six years (may be extended)
- Average annual tailings output 280,000 t
- Plant availability 91.3% (8,000 hrs/yr)
- An average slurry production of 35 tph
- Stage 1 for two years to store 330,000 t of dry tailings
- Stage 2 for Year 3 to Year 6 to store a total up to 3 Mt of tailings
- Thickened tailings pumped at 50% solids with flocculant addition
- Tailings settled density of approximately 1.45 t/m3 (Stage 1)
- Tailings consolidated density of approximately 1.6 t/m^3 (Stage 2)
- Tailings settled permeability 1.0 x 10-6 metres per second (“m/s”). Permeability may decrease further as the tailings consolidate
- Tailings solids specific gravity 3.1
- TSF Stage 1 crest is elevation 234.0m with the spillway at elevation 222.3 m for 11.7m freeboard which includes spillway for 1 in 500 intensity storm event, freeboard and 2m wave and 2m evaporation. Freeboard allowance for Stage 2 is 9 m, comprising 2 m minimum water cover, 2 m evaporation allowance (4 m total water cover) and 5 m spillway depth
- Freeboard allowance for the PSRD is 2 m

18.1.3 Site Roads and Hardstand

Site roads link the pads on the plant site, the underground mine portal and the TSF. It is not cost-effective for a short-life project to develop cut side slopes with long-term stability aiming to maintain access under all weather conditions. Therefore the roads on the site will be cut with steep side slopes. Experience on site from the exploration phase is that while these cut slopes may fail from time to time and bury sections of the road they can be cleared quickly with no long-term adverse effects.

18.2 Off-Site Infrastructure

18.2.1 Accommodation Camp, Messing and Facilities

The camp is located 3 km from the mine site on the east bank of the Chen Stream downstream of its confluence with Dam Creek, on a site already acquired by BPNM. The site is 35 m to 100 m wide and approximately 300 m long. Due to the limited area available, two storey accommodation blocks are proposed of concrete and brick construction, with five rooms on each level. This is in line with building practices in the area.

Accommodation for senior personnel is in single rooms, each with ensuite bathrooms. Allowing provision for contractors, consultants and visitors, five blocks are required. An additional block to the same standard has been included for VIP visitors.

Accommodation for intermediate and junior personnel will be in rooms shared by two people, each with ensuite bathroom. With provision for contractors, consultants and visitors, eight blocks are required.

The standard of accommodation reflects that required to attract and retain appropriately capable personnel for these roles. To minimize the additional costs associated with the construction camp, rooms may be shared during the construction period.

Personnel are grouped in rooms, levels and blocks by gender for privacy, and by department, section and roster to minimize disruption if one group is required to work extended hours and returns to camp when others are already sleeping.

Common use facilities include:

- mess hall, kitchen, cold room and freezer facilities;
- commercial laundry facility;
- wet mess and covered recreation area (pool/snooker/billiards, table tennis);
- dual use training room (with computers for afterhours internet and e-mail access);
- multi-purpose outside tennis, badminton and volleyball court; and
- weight training gym area.

18.2.2 Site Access Road

Site access is via a new all-weather road from the provincial road that parallels Chen Stream. The access road is 8 m wide with a 1 m shoulder on each side, giving a 10 m wide permanent roadway corridor. The maximum incline is 1:8 and the minimum inside radius is 8 m to allow conventional flatbed trucks and prime movers with single semi-trailers to access the site.

This results in a road route on the north bank of West Ban Phuc Creek rising up the valley to the process plant site. Although most of the roads on the site will be cut with steep side slopes, the site access road has two of the site roads and utility corridors located immediately upslope for most of its length.

There is a local airport at Hat Lot, near Son La, which is closed for an upgrade and runway extension to make it jet capable. However, the local airport can be used for helicopter for medical evacuations and other emergencies.

18.2.3 Community and Social Infrastructure

An allowance has been made in AMR's operating budget for community and social infrastructure. In June 2012, AMR constructed a suspension bridge for pedestrian and motorcycle traffic over Soui Chen (or more commonly referred to as Chen Stream) at a cost of approximately US$ 100,000. Completion of this project has enabled villagers from the far side of Chen Stream to gain better access to the communities and amenities along Highway Number 37.

Further community support will be undertaken in consultation with the Moung Khoa Commune and village leaders. Although it is not yet clear what will be most appropriate it is expected that BPNM will be required to contribute to additional school, medical or other infrastructure facilities or upgrade of existing facilities either as sole contributor or on a participatory basis with the local, provincial or national governments or a combination of agencies. AMR will also seek to leverage its contributions with international aid organisations.

18.2.4 Transport

Some supplies are available at Son La, capital city of the province of the same name about 55 km northwest of Ban Phuc. A 2,000 megawatt ("MW") hydroelectric dam is located 130 km downstream from Ban Phuc at Hoa Binh and supplies power to the national grid. The Hoa Binh Dam blocks the Da River, and forms a lake extending 200 km upstream which, when navigable, is used for barge transport of product from and supplies to the project site. However during the dry season there may be insufficient depth of water at Ban Phuc for barge access. Therefore direct trucking of concentrate from and supplies to site has been selected by BPNM. Concentrate produced will be trucked in containers.

Incoming freight will consist of all manner of equipment, spares, reagents, consumables, tyres, fuels, lubricants, food and general merchandise. Some inbound goods are in break bulk but others are in 20 foot sea containers. Road transport of diesel fuel is in conventional tanker trucks.

A quotation has been received for the concentrate transport system including provision of facilities and equipment and operation for delivery to the Port of Hai Phong, the major northern port servicing Hanoi and north-western Vietnam.

Toll International Pty Ltd ("Toll") identified a number of routes for road transport of concentrate from and goods to site with the size of the loads being the main determining factor as to which route is taken. As most transport will be to and from Hai Phong the routes pass through Hanoi and Hoa Binh and are simply reversed for return journeys; the description is in the context of goods delivered from Hai Phong to site:

- All "in gauge" loads and "out of gauge" container and conventional loads up to 3 m cargo height will travel on Highway 5 from Hai Phong to Hanoi, Highway 6 from Hanoi to the intersection with Highway 37 and then on Highway 37 to site.
- All "out of gauge" loads over 3 m cargo height will travel along Highway 10 from Hai Phong and then on to Highway 18 to Hanoi where the road links with Highway 6 and continues on to site as described above.

18.2.5 Concentrate Transport

Toll have provided recent advice and cost estimates for in-country transport and handling of project concentrate.

The Project will produce up to 70,000 tonnes of concentrate per annum. The concentrate will be loaded into modified sea containers at site and transported to road to Hai Phong harbour for shipment to the Jinchuan smelter in China.

Once loaded on board responsibility for the concentrate will be transferred to the concentrate buyer who will absorb insurance and transport obligations.

18.2.6 Power Supply and Distribution

A 35 kilovolt power transmission line runs from the Son La sub-station, some 40 km from the Project, to within 1 km of the Project. The Son La Provincial Government Power Department (PC Son La) has submitted a design for a 1 km spur line to the proposed 35kV/6.3kV substation. Power will be reticulated to the process plant, tails line pump stations and the underground mining operations through low voltage motor control centres (MCC). MCCs and transformers and switchgear for the substation have been ordered.

18.2.7 Communications

The Project is connected to the national grid via a fibre-optic connection. This provides telephone, facsimile links and broadband internet access. Internally, the camp and process plant site are connected with wireless LAN coverage.

Two-way radio communications are established with fixed, in-vehicle or hand-held units accessible to all personnel.

18.2.8 Raw Water Supply and Potable Water

Process water will be recycled from the tailings storage facility (TSF) with make-up and raw water drawn from the Chen Stream which feeds into the Da River.

The camp will draw water from the Da River to supply a reverse osmosis plant for domestic, non-potable water use.

Drinking water is provided in bottles or purpose made containers.

18.2.9 Wastewater Treatment and Disposal

Packaged sewage treatment plants is located at the camp site and is to be installed at the process plant site.

Solid waste will be disposed as appropriate in landfill within the project area, in stopes being backfilled, the tailings dam or by removal from site by contractor.

18.2.10 Fuel Storage, Dispensing and Waste Oil Disposal

The main fuel facility is located adjacent to the warehouse. One 98 m3 storage tank will provide about 30 days storage capacity for diesel. The tank will be contained within an earthen bund. Road transport of diesel fuel will be in conventional tanker trucks. Diesel will be transferred to the tank at the warehouse and distributed around the site by a mine service vehicle. The dispensing facilities will include a metered bowser and areas drained to oil traps for vehicle refuelling.

Waste oil from oil traps and drained from mobile and fixed equipment will be collected and transferred into drums contained within the bund at the fuel facility for removal from site by contractor.

19 Market Studies and Contracts

19.1 Market Studies

Once operational, the Ban Phuc Nickel Project will produce a mixed sulphide concentrate containing nickel, copper and cobalt. All of the concentrate is to be sold to Jinchuan Group Ltd (Jinchuan) under the offtake agreement entered into between BPNM and Jinchuan in 2008.

Nickel, copper and cobalt are exchange traded metals and the pricing terms BPNM's offtake agreement are linked to London Metal Exchange quoted prices. As such, no market studies are intended to be undertaken.

19.2 Contracts

19.2.1 Offtake agreement

BPNM entered into an offtake agreement with Jinchuan on 28th April 2008. Jinchuan agreed to purchase all nickel concentrates produced during the life of the initial Ban Phuc Project. The agreement also granted Jinchuan first refusal option on additional nickel concentrates that BPNM may produce from new projects other than Ban Phuc.

19.2.2 Power Supply

BPNM is currently working on a power supply agreement with Son La Power Company, the provincial power company, for the supply of electricity to site. A Letter of Intent has been issued to Son La Power Company.

19.2.3 Engineering

BPNM has entered into a contract with Aurecon (Vietnam), a subsidiary of an international engineering firm, for engineering design associated with power and reticulation, and Knight Piesold, an international engineering firm, and CCDC, a Vietnamese engineering firm, for the design of the TSF.

19.2.4 Construction

AMR is progressing construction as an owner-build project. As such there is no EPCM contractor associated with the project. On 26 April 2010 BPNM entered into contracts with Coninco, a Vietnamese engineering firm for the oversight of construction and overall quality control associated with the project construction.

19.2.5 Equipment

All major items of equipment associated with the project have already been purchased.

19.2.6 Transport

Contracts for the transport of concentrate from site to Hai Phong have not been finalized.

19.2.7 Laboratory contract

A Tender for the establishment of an independent onsite assay laboratory is currently being prepared, but at the time of writing have not yet been issued.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

This section provides an overview of AMR environmental management and community commitments for the Ban Phuc Nickel Project.

AMR operates on the philosophy of developing a sustainable business through an effective environmental management system to provide adequate and suitable protection to the environmental envelope they operate within.

AMR's engagement with the local communities is guided by four broad principles, as defined by their Corporate Social Responsibility Policy. These four principles are:

- Respect the cultures, customs and values of individuals and groups whose livelihoods may be affected by exploration, mining and processing;
- Recognise local communities as stakeholders and engage with them in an effective process of consultation and communication;
- Contribute to and participate in the social, economic and institutional development of the communities where operations are located and mitigate adverse effects in these communities to the greatest practical extent; and
- Respect the authority of national and regional governments and integrate activities with their development objectives.

20.2 Local Legislative Requirements

20.2.1 Environmental Legislative Requirements

The general basis of environmental regulatory development in Vietnam is driven by two key framework documents, namely

(i) *The National Plan for Environment and Sustainable Development in 1991: A Framework for Action in 1992*; and

(ii) *National Strategy on Environmental Protection towards 2010 and Orientations towards 2020*. The *National Plan for Environment and Sustainable Development in 1991* provides the strategy for the sustainable management of Vietnam's natural resources. In December 2003, the Vietnamese government approved the *National Strategy on Environmental Protection towards 2010 and Orientations towards 2020*. The 2003 National Strategy sets targets to be achieved, including reducing pollution and improving the environment quality.

20.2.2 Environmental Impact Assessment (EIA)

The Law on Environmental Protection (LEP) 2005 (No. 52/2005/QH11LEP) is the overarching legislation for environmental management and protection in Vietnam and Article 18 of the LEP, 2005, requires an Environmental Impact Assessment (EIA) be prepared for projects involving "clay and stone exploitation and processing", and is thus considered a relevant legislative control to the project. Other relevant decrees and circulars include:

- Decree No. 29/2011/ND-CP of April 18, 2011 provides guidance on the environmental approval process and related documents to be prepared in order to satisfy Article 18 of the LEP 2005;

- Circular 08/2006/TT-BTNMT issued by Ministry of Natural Resources and Environment (MONRE) provides further clarifications on the contents of required reports and documents, including strategic environmental assessment, EIAs and environmental protection commitments; and

- Circular No. 26/2011/TT-BTNMT issued on 18th July 2011 requires a new EIA report be submitted for any project which has not been implemented 36 months after initial EIA approval, or when there are changes to the project which may impact total waste generation.

The first stage of the environmental approval process is to submit a Strategic Environmental Assessment (SEA). This allows for project screening and classification. Projects are classified into either Category One (1) or Two (2) for assessment, with guidance on the requirements for the next stage provided by appropriate authorities. An Environmental and Social Impact Assessment (ESIA) is usually required for Category One (1) projects to address the current state of the environment in the project area, impacts on the environment, any proposed mitigation measures for environmental protection, and other more general technical information on the Project.

Following the completion of the ESIA, the ESIA is reviewed by the MONRE and the Department of Natural Resources and Environment (DONRE). The evaluation committee usually comprised of seven to nine members from various Government departments and Institute or Environment, Science and Technology (INEST). The ESIA would then be either (i) approved, (ii) approved with conditions; or (iii) not approved. Once approved, the proponent is expected to implement the conditions listed in the submission document. Figure 82 shows the typical ESIA approval process in Vietnam.

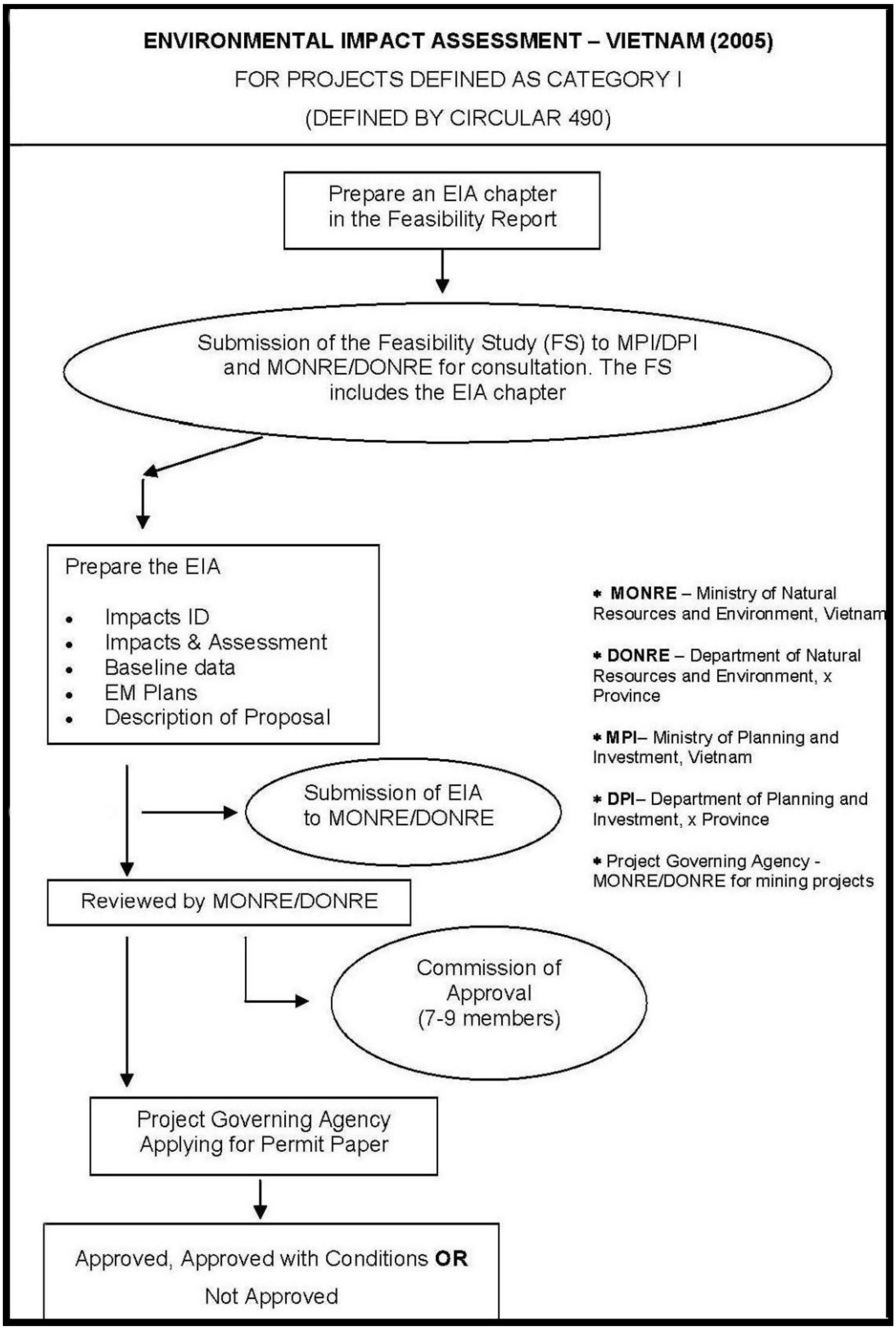


Figure 82 Typical ESIA Approval Process in Vietnam

20.3 Other Relevant Environmental Requirements

Other environmental clauses relevant for the Ban Phuc Nickel Project include:

20.3.1 Environmental Impacts Monitoring and Reporting

Article 94 specifies for environmental impact monitoring program to track changes in quantity, composition and toxicity of solids wastes, gases and wastewater.

Circular 08/2005/TTBTNMT provides guidance on the contents of the environmental monitoring program that need to be established while Circular 12/2006/TT-BTNMT (dated 26th December 2006) establishes bi-annual reporting for hazardous waste generation and management to DONRE.

20.3.2 Environmental Emissions Standards

Article 8 of the LEP 2005 provides for the formulation and application of environmental standards to meet the environmental protection objectives, preventing environmental pollution, degradation and incidents. A number of relevant Vietnamese Standards have been formulated:

- Air Emissions: QCVN 19-2009/BTNMT industrial Emission of Inorganic Substances and Dusts and QCVN 20:2009/BTNMT Industrial Emission of Organic Substances;
- Wastewater Discharge Quantity: TCVN-2005 for Industrial Wastewater Discharge Standard. Decree 149/2004/ND-CP requires that a project owner must obtain a permit to use water resources and discharge wastewater into water courses;
- Solid and Hazardous Waste: Decree No 59/2007/ND-CP Solid Waste Management and Circular No 12/2006/TT-BTNMT Guiding the Practice Conditions, Procedures for Compilation of Dossiers, Registration, Licensing of Practice and Hazardous Waste Management Identification Numbers;
- Soil and Groundwater: QCVN 09-2008/BTNMT for Groundwater Quality, QCVN 24-2009/BTNMT for Industrial Wastewater Quality, QCVN 03-2008/BTNMT Heavy Metals Concentrations in Soil and QCVN 03-2008/BTNMT Allowable Pesticide Residues in Soil;
- Surface Water Quality: QCVN 08-2008/BTNMT Surface Water Quality; and
- Noise and Vibrations: QCVN 26-2010/BTNMT National Technical Standard for Noise and TCVN 6962-2001 Allowable Vibration and Shock from Construction Activities and Industrial Production.

Other potentially related environmental protection related legislation includes the Law on Water Resources 1998, Cultural Heritage Law 2001 and Biodiversity Law 2008, each of which is summarised below:

- Law on Water Resources 1998

The Law on Water Resources prescribes the management, protection, exploitation and use of the water resources. Article 5 of the Law on Water Resources requires business operators to apply for permits associated with the use and discharge of water. The objective is to regulate the discharge of wastewater into water sources and prevent environmental pollution.

- Cultural Heritage Law 2001

Organizations and individuals involved in a development have responsibility to apply measures for the protection and promotion of cultural heritage under the Cultural Heritage Law 2001.

- Biodiversity Law 2008

The Biodiversity Law 2008 provides for the conservation and sustainable development of biodiversity and the rights and obligations of organizations, households and individuals in the conservation and sustainable development of biodiversity. All activities carried out within conservation zones and buffer zones of conservation zones shall be accompanied by an EIA, and activities need to be managed to prevent adverse impacts on the conservation zones.

20.3.3 Mine Closure Requirements

Article 44 of the LEP 2005 and Article 47 of the Law on Minerals 2010 provide specific environmental protection and rehabilitation requirements after the completion of mineral prospecting, exploitation and processing activities.

The requirements of mine closure in Vietnam are defined in the Decision 1456/QD-DCKS. A mine closure plan is to be submitted to the PPCs for approval and shall include at least the following information:

Reason for mine closure;

- Work tasks and measures of mine closure, including necessary works to ensure safety of the mine after closure (i.e. waste dumps, rehabilitation of related land and environment);
- Recommendations on proposed use after mine closure and rehabilitation schedule; and
- Cost of mine closure, and sources of funding including details on any compensation that is required to be paid at the time of mine closure.

Mine closure schemes must be appraised and approved in terms of contents and on safety, environment, land rehabilitation and other requirements.

In addition, Decision No 71/2008/QDD-TTG dated 29 May 2008 requires that all organizations and individuals with permits to explore for minerals, leave a cash deposit for environmental rehabilitation (or more commonly referred to as the Rehabilitation Security Deposit) into Environmental Protection Fund. The Vietnam legislation (Decision No 71/2008/QDD-TTG) provides the methodology for determining the Rehabilitation Security Deposit. The Rehabilitation Security Deposit is deposited with MONRE who provides oversight for approving, inspecting and certifying the environmental rehabilitation and restoration requirements.

Rehabilitation Security Deposit requirements for the Ban Phuc Nickel Project is discussed in Section 20.15.14.

20.3.4 Involuntary Resettlement Legislative Requirements

The process of involuntary resettlement is governed by the Land Law 2003 (No. 13/2003/QH11) and supported by a number of other decrees and circulars:

- Decree 181/2004/ND-CP (29 October 2004) guiding the implementation of Land Law 2003;
- Decree No. 17/2006/ND-CP (27 January 2007) on amendment and supplementation to a number of Articles or Decrees guiding Law on Land and Decree No. 187/2004/NF-CP;
- Decision No. 153/2004/QD-TTg of the Prime Minister (17 August 2004) promulgating the strategy for sustainable development in Vietnam (Vietnam Agenda 21);
- Decree No. 197/2004/ND-CP (3 December 2004) on compensation, assistance and resettlement when land is recovered by the State and Circular No. 116/2004/TT-BTC (7 December 2004) guiding the implementation of December 3, 2004 Decree;
- Decree No. 188/2004/ND-CP (26 November 2004) on methods of determining land process and assorted land price brackets and Circular No. 114/2004/TT-BTC (26 November 2004) guiding the implementation of Decree No. 188/2004/ND-CP;
- Decree No. 198/2004/ND-CP (3 December 2004) and Decree No. 142/2005/ND-CP (14 November 2005) on collection of land fee and rental respectively;
- Decision No. 134/2004/QD-TTg of the Prime Minister on assistance policies to Ethnic Minorities; and
- Decisions coming from the Provincial and PPCs.

20.3.5 Permitting and Approvals

AustralAsian Resource Consultants Pty Ltd (AustralAsian) completed an ESIA in September 2005 as part of the Feasibility Studies for the BPNM. The ESIA was subsequently updated by Centre for Environment Consultancy and Protection (CECP) to satisfy the requirements of the LEP 2005 and other relevant environmental and social related Vietnamese legislation outlined above.

The ESIA provides environmental and social baseline information and assess the potential impacts on the bio-physical and socio-economic environment and covers construction, operational, closure and rehabilitation phases of the project. Affected communities were consulted and their concerns reflected and addressed within the ESIA. A number of mitigation measures associated with each phase of the project were recommended.

The following permits have already been obtained by AMR:

20.3.6 Environmental Approvals

- Environmental and Social Impact Assessment (reference no. 1942/QDBTNMT) (ESIA), granted 25 December 2006;
- Forestry Approval (reference no. 1133/BNN-LN), granted February 2007;
- Rehabilitation Security Deposit and Mine Closure Plan (reference no 1859/QD-BTNMT), granted October 2011; and

20.3.7 Resettlement Approvals

- Final Resettlement Plan, approved 2007.

20.3.8 Other Permits

Under the Vietnamese regulatory permitting system, a number of other permits are required for the operation of the AMR. Permits for Discharge of Wastewater, Water Use, and operation of the Water Collection Dam are currently pending submission. AMR plans to submit the permits by April 2013.

20.4 Community Resettlement

Two separate stakeholder consultations phases were undertaken by AMR over the last half of 2004 at various locations potentially affected by the BPNM Project. Consultations were conducted in the form of face-to-face meetings at the various villages, at village meetings and at the Muong Khoa Commune Office. Relevant stakeholder groups include:

- Government agencies, such as MONRE, Ministry of Planning and Investment, Ministry of Water Resources, Ministry of Construction, Ministry of Energy, Ministry of Agriculture and Rural Development, Ministry of Industry and Trade and Son La Electricity;
- Local commune councils including the Muong Khoa and Bac Yen District Government, Provincial Government of Son La Province;
- Village representatives from affected communities; and
- Representatives from AMR.

Throughout the development of the ESIA, AMR maintained constant communications with villagers, district and provincial government representatives through regular attendance at meetings. The villagers and district and provincial government representatives are kept up-to-date with project development (including clarifying any misconceptions).

AMR also commissioned a household baseline survey to obtain relevant socio-economic data, including land tenure, livelihood, demographics, etc. These data were utilised by AMR for assessing potential socio-economic impacts on the villagers and develop suitable mitigation measures.

The Vietnam Government subsequently approved a Resettlement Action Plan in 2007. The Ban Phuc Nickel Project requires the resettlement of 111 households and ten organizations in total. AMR has separated the resettlement process into Phases 1,2 and 3, the status of which is summarized below:

20.4.1 Phase 1 Resettlement

The Phase 1 resettlement was completed in 2004. A total of 11 households were moved to different locations around Son La Province. The decisions made in consultation with the Vietnamese government for Phase 1 Resettlement are summarized below:

- Son La PPC's Decision Ref No 1810/QD-UB (2004), confirming the withdrawal of 67,567 m^2 of land from organizations and individuals based in Muong Khoa commune, Bac Yen district to allocate land use for offices and landing ground;
- Muong Khoa Communal People's Committee (CPC) Decision, concerning the establishment of the Land Clearance and Compensation Commission to coordinate the construction of BPNM's River Port and office buildings;
- Bac Yen DPC's Decision No 1047/QD-UB, approving site clearance compensation value applicable to the Muong Khoa commune; and
- Agreement to the Compensation Plan by Muong Khoa commune which provides breakdown of the compensation for property and crops in Pom Don Hill, Ban Pho village and Ban Khoa Market area.

20.4.2 Phase 2 Resettlement

The Phase 2 Resettlement was completed in 2007 and covers over 100.07 ha of the mine site area. This area incorporates the mining area, plant site, TSF, internal roads, other facilities. The 100.07 ha include land belonging to the Ban Khoa village (used for construction of TSF) and Ban Phuc village (used for building of the mining area, plant, roads and other facilities). A total of 89 households and two organizations were resettled.

Decisions made in consultation with the Vietnamese government for Phase 2 Resettlement are summarized below:

- Decision No: 24/ QD-UBND of Bac Yen DPC dated 15/01/2008, and Decision No : 56/ QD-UBND issued by Bac Yen DPC dated 23/01/2008 regarding compensation and resettlement for 57 households and Ban Khoa cooperative at Ban Khoa Village (relating to acquisition of land for TSF);
- Decision No: 2166/ QD-UBND of Bac Yen DPC dated 21/11/2008 , and Decision No : 2152/ QD-UBND issued by Bac Yen DPC dated 16/11/2007 regarding compensation and

resettlement for 32 households and Ban Phuc cooperative at Ban Phuc Village (relating acquisition of land for building of the mining area, plant, roads and other facilities); and

- Decision No: 696/ QD-UBND of Son La PC dated 19/3/2009 allowing BPNM to use 100.07 ha for mining purpose.

Final Resettlement Action Plan (Table) – The Phase 1 (22 households and 2 organizations) and Phase 2 resettlements (89 households and 2 organizations) were completed in 2004 and 2007 respectively. The Phase 3 resettlement commenced in 2007 and is currently near completion. The total compensation payout for the Phase 3 resettlement was approximately US $36,200 and has been paid out in November 2012.

20.5 Stakeholder Engagement Philosophy

AMR currently maintains open communications with stakeholders through consultations with the regulatory authorities and communities, as and when required. AMR plans to adopt similar strategy as the mine continues through construction phase, and will continue to do so into the future (i.e. during operational and closure phases). AMR will endeavour to continue disseminate accurate information from time to time when the need arises, and foster open communications between AMR and the relevant stakeholders. AMR is currently developing a grievance mechanism protocol to formalize procedures for interaction with the stakeholders.

20.6 Environmental Baseline Data

This section summarises general environmental baseline data collected during the ESIA, including meteorological information, land and water conditions, flora and fauna, air and noise quality.

20.6.1 Meteorology

The project area climate is dominated by tropical monsoon weather conditions that affect all of mainland Southeast Asia, and produces distinct wet and dry seasons. The area is dominated by very hot summer months with high levels of rainfall, and a cold winter with little rain.

The majority of the rainfall falls during the wet season, typically between April and September; the average maximum rainfall received in the wet season is up to 250 mm per month. Due to the influence of isolated storms and tropical low pressure systems this figure may be significantly higher in some months. The dry season is typically between October and March. Average annual rainfall is 1,289 mm per year.

Evaporation rates within the project area are highest during February to April ahead of the wet season. The evaporation rate for the project area ranges between 52 and 137 mm per month. The average evaporation rate for the area is approximately 1,086 mm per year, indicating a net water surplus for the Project.

Between 1999 and 2003 monthly average temperatures ranged between 14.4 ^{0}C and 25.7 ^{0}C. The maximum temperature was 35.9 ^{0}C the mean was 24 ^{0}C, and the minimum temperature was 0.7 ^{0}C.

The average humidity within the project area is 79%. Humidity is higher in the rainy season (May to October) and is often greater than 82%.

Available data indicates that prevailing winds are from a south-easterly direction, during the period October to April. In the first half of the wet season winds are from the northwest and southeast. North-westerly winds predominate in the second half of the wet season.

Further details on climate are discussed in Section 5.3.

20.6.2 Topography

The Ban Phuc deposit is located within rugged terrain in mountainous areas in the north-west of Vietnam. The topography of the project area ranges between steeply sloping hills with elevations between 100 and 550 m above sea level and narrow valleys with few flat areas.

Further details on topography are discussed in Section 5.1.

20.6.3 Soil Types

Soils in the Ban Phuc area vary from alluvial clays and loamy soils in the lowlands to reddish-brown dermosols and ferrosols in the upland regions. Upland soils are very easily eroded once they have been cleared of vegetation for agriculture. The upland soils are of low to moderate fertility. Lowland soils exhibit a moderate to high clay content and are also of low to moderate fertility.

20.6.4 Land Use

The most significant land use in the project area is forestry, which comprises mixed growth, mature forest and shrub forest. Land use is reported to be (i) agriculture (22%), (ii) forestry (69%), (iii) inhibited (2%) and (iv) others (7%), also rice (paddy and upland), upland corn and aquaculture.

20.6.5 Natural Hazards

Natural hazards in the vicinity of the site include weather events (i.e., typhoons and heavy rainfall mentioned above, earthquakes and forest fires). During the period between 1973 and 2004, a total of 28 seismic events with a magnitude greater than 5.0 on the Richter scale have occurred within 500 km of the project area. Of these events, only three occurred within 200 km of the project area. Since 1973, only four seismic events have occurred within 100 km of the Project, all of which were recorded to be less than 5.0 on the Richter scale.

The main cause of forest fires in Vietnam is the use of slash-and-burn techniques associated with land clearance and shifting cultivation between March to May.

20.7 Water Resources

20.7.1 Surface Water

Surface water run-off from the Ban Phuc deposit and the immediate surrounds flows into four small creek lines, tributaries to Suoi Chen (Chen Stream) which flows in a northerly direction to where it meets with the Song Da (Black River). The Da River flows east-southeast to Hoa Binh where it turns to the north and joins the Song Hong (Red River) near Viet Tri.

The majority of the small tributaries within the project area are seasonal and generally have little to no flow during the dry season and good flows during the wetter months. The Da River has been dammed for hydroelectric power generation at Hoa Binh and more recently at Son La, approximately 50 km upstream of the confluence of Chen Stream and the Da River. The flow in the river is controlled to maintain the level in the dams.

20.7.2 Surface Water Quality

A number of surface water samples were collected as part of the ESIA baseline data process and assessed against the Vietnamese standards. The surface water quality results indicated that the pH and majority of the heavy metals were below detection limits and within the Vietnamese regulatory standards. Total Suspended Solids (TSS) concentrations were high in the surface water samples taken during the rainy season and ranged between 166 mg/l and 1,415 mg/l. This exceeded the Vietnamese standard (acceptable limits between 6 mg/l and 26 mg/l). However, this is likely to be related to natural conditions (i.e. steep terrain) and potentially deforestation.

20.7.3 Groundwater

There are year-round groundwater discharges into streams, namely Ban Phuc Creek and Dam Creek, indicating reliable groundwater supplies. Groundwater levels vary seasonally with the lowest levels experienced in winter and the highest in summer.

20.7.4 Groundwater Quality

The pH of the groundwater generally varies between 7.0 and 8.5 and was within the Vietnamese groundwater standard. The concentrations of heavy metals (Nickel, Iron, Lead, Chromium, Copper, Zinc and Manganese), cyanides and sulphates were also below Vietnamese groundwater standard. The hardness of the groundwater tested as part of the ESIA ranged between 43 mg/l and 442 mg/l and exceeded the Vietnamese groundwater standard but appears to be a result of natural geological conditions.

20.7.5 Domestic Water

Most villages source water from streams and groundwater. The percentage of users of water from streams varies between villages but averages over 90%. The percentage of users utilizing wells and groundwater is lower depending on the village but tends to be less than 10% of the population.

Water based paddy rice production and the farming of fish in ponds by people who live adjacent to streams are both significant consumers of water in the project area.

20.7.6 Drinking Water Quality

Samples of potable water were taken from two typical domestic households in the area and also from the Ban Phuc and Ban Trang water supplies. The overall results from households and water supply sources indicated that the quality of domestic water is relatively high and, with the exception of excessive coliforms and iron detected above the Vietnamese standard.

20.8 Flora and Fauna

The Project is located within mountainous country that contains areas of relatively degraded and localized deciduous forest, providing an important habitat for a range of fauna species. The dominant ecosystem is the bamboo forest, with re-growth secondary forest also identified within the project area. The flora and fauna in the project area currently present no material issues for the project.

20.8.1 Flora Species

A total of 464 plant species were identified during the survey, and are further classified into 325 genera, 113 families and 5 vascular phyla. The species found on site belong to 16 geographic elements out of a possible 20 geographic elements of Vietnamese flora. The Indochina element is the richest in species, incorporating 119 species and making up 25.6% total species of the flora. The project area maintains five endemic, rare and valuable species as listed in the Vietnamese Red Book of Endangered Species or in the International Union for the Conservation of Nature and Natural Resources Red List of Threatened Species. A number of useful vegetative species exist in the project area, including medicinal (38.3%), timber (20%) and food (18.7%).

20.8.2 Fauna Species

The fauna survey conducted during the ESIA indicated that a total of 13 mammal species from 10 genera and 7 families were reported to occur in the Muong Khoa area, including the Rhesus Monkey (a rare and vulnerable mamma). A total of 31 bird species and of 9 amphibian species were identified within the project area. A further 10 species of reptiles were also identified, with six of these being listed in the Vietnamese Red Book as being either threatened or vulnerable.

A total of 36 species of fish were identified as occurring or potentially occurring in the streams and rivers of the project area. The survey identified 51 species of phytoplankton belonging to three algal-phyla and 28 plankton species. Sixteen benthic species were identified in the project area, with an average density of species of 17.3 animals/m^2.

20.9 Air Quality

At each of the villages adjacent to the Ban Phuc Nickel Project, the majority of parameters assessed were within Vietnamese national and provincial standards for the 24-hour and the one-hour samples. Nickel levels exceeded both the 24-hour and the 1-hour standards by a significant amount and have been attributed to the high levels of natural background nickel in the soils surrounding the project area.

20.10 Noise

Almost all of the ambient noise levels recorded at the sampling sites were below the 60 decibels (dBA) level specified in the relevant Vietnamese criteria (60 dB(A) for LA_{eq} between 0600 hrs and 1800 hrs, 55 dB(A) for LA_{eq} between 1800 hrs and 2200 hrs, and 45 dB(A) for LA_{eq} between 2200 hrs and 0600 hrs). The average LA_{eq} (one-hour) values were higher in areas closer to Ban Phuc Village and National Highway No. 37.

20.11 Socio-Economic Baseline Data

Information was collected through the use of district government censuses, the statistics yearbooks of the Son La Province and the Bac Yen District, relevant archives of the Muong Khoa Peoples Commune and two household surveys commissioned by AMR.

20.12 Demographic Profile

The majority of the population in the project area live within clearly defined village settlement boundaries, predominantly along National Highway No. 37. In total, the five villages within the survey area comprise 977 individual households with a combined population of approximately 5,029 people with an average household size of 5.2. The size of each individual village varies with Ban Phuc and Ban Khoa located along Chen Stream being the largest.

Population distribution is dominated by people in the 5 - 9 year old and the 15 - 19 year old age brackets. Figures indicate the relatively long life expectancy and young composition of the project area.

20.12.1 Ethnicity

The Muong Khoa commune is essentially divided into three specific village zones (i) 11 villages in the highland zone, (ii) 5 villages in the riparian valley, and (iii) 3 villages along Da riverbed.

The population within the project area is characterized by five ethnic groups, namely Thai, Hmong, Muong, Kho Mu and Kinh groups. Of the total population, the Thai and Muong make up the majority, with household sizes in the highland zone typically larger than in the lowland areas.

20.12.2 Housing

Housing within the project area varies depending on the ethnicity of the household. The predominant building materials include timber and woven bamboo and a household survey identified that the majority (66%) of houses are solid or semi-solid structures. Only 64% of the residences have access to electricity and fewer (45%) have access to clean water.

20.12.3 Food Resources

The main food source in the area is based on subsistence farming. Income sources for the people of Muong Khoa are based on agriculture and forestry. Food security in the commune area has recently improved with annual per capita food production reaching 802 kg. Rice cultivation is extensive within the project area and the majority of villages harvest one seasonal crop of paddy rice as little irrigation is used. Other crops grown in the area within and surrounding the Project include vegetables, corn, taro, cassava, soy bean and sugarcane.

Animal production is another primary activity with a variety of domestic species used for food (pigs and poultry), transportation (horses) and ploughing (oxen and buffalo). Fish provide a significant contribution to the diversification of the local economy and the food requirements of the area.

20.12.4 Health

Key findings of a preliminary health and nutrition assessment of the project area are as follows:

- There is one government run health clinic within Muong Khoa with 19 beds and a staff of 4 physicians and 3 nurses (2 of which are midwives);
- Of all 19 villages in the commune, 16 have their own government health station, signalling a relatively good network of community health care in the area;
- A vaccination program was present in all 19 villages;
- The program against child poverty was developed in 2004;
- The majority of the households in the commune enjoy free commune health insurance for basic diseases and health access;
- The hygiene and sanitary status of the project area is moderate to poor with only 45% of houses having access to clean water; and
- The most common forms of illness episodes within the district during 2003 were diarrhoea and pneumonia.

20.12.5 Education

The level of formal education within the project area is reasonably high with 100% of those surveyed in the household survey are reportedly literate. Nearly all men and women have gained a minimum of a primary education. Of the surveyed villages, 95% of all school age children in these villages have gained a primary level education. A primary and secondary school exists in the Muong Khoa commune and offers a variety of classes. Villagers also have high level of skills that have been acquired for subsistence living, such as hunting, gathering, farming, fishing and weaving.

20.12.6 Traffic

Over the last four years, road conditions within the region have steadily improved with the number of villages with vehicle access increasing from 8 in the year 2000 to 14 in 2004. Based on the ESIA report, the most common forms of transport within the project area are cars and motorbikes and this reflects the growing use of motorized vehicles in Vietnam. Traffic density within the region is not regarded as high.

20.12.7 Archaeology

There are no cultural or historical sites except for a monument to revolutionary martyrs in Ban Trang Village, based on surveys conducted during the ESIA. The Muong Khoa people have also affirmed that they do not have any relics in their possession and that there are no other relics to be found in the area.

20.13 Potential Environmental and Social Impacts and Proposed Mitigation Measures

The following potential environmental, social and health and safety impacts were identified relating to the construction, operational and closure phases of the project:

20.13.1 Land Resources

The BPNM Project will result in the clearance of approximately 45 (Table 65) ha of land, consisting of mostly regenerated forest and riparian vegetation.

Table 65. Areas of Land Potentially Impacted

Project Site	Land Use	Potentially Impacted Areas (Ha)
Process Plant, RoM pad, access roads, sediment trap, TSF, tailing slurry pipeline, site roads	Forests along river/stream	15
	Vegetated forest	30
	Total	45

There will be in-situ loss of soil resources beneath the mine infrastructure, breakdown of soil structure through compaction, loss of soil from cleared areas due to water runoff and reduction in fertility. The following land clearance protocols have been deployed by AMR:

- Prior to land clearance, any commitments such as the salvage of timber and vegetation, the flora/fauna inspection of the site and the potential unearthing of archaeological artefacts are to be completed and signed off by the site supervisor;
- Vegetation located within the proposed disturbance areas should be cleared and windrowed to outside the disturbance footprint. The windrowed material will be left in situ to provide habitat for fauna; and
- Following vegetation clearance, the area will be ripped to allow dozer blades or scrapers to work efficiently. Where possible a minimum of 200 mm of topsoil

will be stripped and stockpiled in designated areas on site to be used in subsequent rehabilitation programs.

Once the land is stripped, the topsoil will be managed in accordance with the following protocols:

- Topsoil and subsoil will be stripped and stockpiled separately;
- Topsoil and subsoil stockpiles will be placed in windrows away from drainage paths and will be no higher than two and three metres in height;
- Stockpiles will be placed in locations close to future rehabilitation sites and will have water diversion structures placed around them to prevent erosion;
- Stockpiles will be allowed to revegetate naturally so as to increase stability and also reduce the likelihood of erosion. If considered necessary, stockpiles will be seeded to promote a cover crop to reduce erosion; and
- A database will be kept on the location, the age and the volume of each soil stockpile.

Diversion banks and channels will be developed to divert surface water around the disturbed areas. The processing plant of the BPNM Project will be visible from West Ban Phuc Valley. When viewed from Noong Oa Hill, the general site activities will be in considerable contrast to the existing rural environmental setting.

There is potential for land contamination to arise from the transport, use and storage of hazardous goods and chemicals associated with the mining operations, including leaching of trace metals, acid drainage from waste rock and/or tailings, process reagent spillage, inappropriate disposal of oily wastes and solid waste materials. Other potential sources of contamination include operational spillages from tailings, slurry, and infiltration of process water from TSF to the sub-surface environment during the project operational phase.

20.13.1.1 Mitigation Measures

The following mitigation measures have been implemented to manage land disturbance from the mining activities:

- Land Clearance
  - Buffer zones will be maintained along creeks and rivers and clearance of vegetation will be restricted to clearly delineated areas; and
  - Site specific survey will be undertaken to identify any listed plan species which may occur in land clearance areas.
- Chemical Management
  - All transport vehicles will have fully sealed floor and covered roof to prevent spillage of soils and materials;
  - Chemical substances will be transported in suitable trucks or barges;
  - Secondary containment will be provided for chemical substances and refilling locations will be restricted;

   - Develop preventive maintenance program for vehicle, heavy equipment and storage to ensure no leakage of fuel and chemical substances;
   - Aboveground storage will be equipped with means of overflow controls; and
   - Spill interception equipment or containment will be used during unloading and loading activities.
- Tailings Slurry and TSF
   - Tailings slurry pipeline will be bunded on either side to prevent environment contamination from tailings slurry in case of leakage or spillage; and
   - TSF will be constructed to prevent infiltration from the dam.

20.13.2 Water Resources

There is potential for contaminated surface runoff (related to sediment and chemical contaminants) from areas disturbed by the project, stockpiles (ore and topsoil), waste rock dumps and tailings to impact upon surface water quality (Table 66). This can lead to increased turbidity and suspended solids in local streams and rivers.

Table 66 Potential Water Contamination Sources

Source	Types of Contamination
Stockpiles (ore and topsoil) and Waste Rock Dam (WRD)	Potential flooding at WRD Drainage containing sediment, dissolved salts, heavy metals, potential acid rock drainage (ARD) and sediment to TSF and creeks
TSF and Tailings Slurry Pipe	Contaminated wastewater to Ban Trang Creek, leach to groundwater and erosion around tailings dam
Process Water	Sediment, dissolved salts, oil, grease, process reagents and heavy metals from the overflow of drains, catch dams or pipe failure Nickel in process water
Sediment Traps	West Ban Phuc Creek overflow Sediment water to Chen Stream
Roads and Infrastructure Areas, ROM Pad	Runoff containing oil, grease, detergents, sediments and reagents/fuel
Sewage	Discharge point, effluent bacteria, organic matter and nutrients

A characterization program was also undertaken by Knight Piesold in 2005 to determine acid forming potential of the waste rocks and tailings. The Knight Piesold assessment noted that waste rocks from the area had low to moderate sulphur content. AMR is currently confirming with Knight Piesold on the requirements for further testing of waste rock samples (i.e. tests to quantify the extent of Net Acid Forming (NAF) and Potentially Acid Forming (PAF) waste rock materials) once the mine is operational.

20.13.2.1 Mitigation Measures

The following mitigation measures have been implemented to manage and/or reduce potential impacts to the water resources in and around the project area:

- Soil and Erosion Controls
  - Slash/debris entering watercourses and drainage channels will be removed. Sediment control measures (including topsoil stockpiles management) will be implemented;
  - Segregate site drainage with routing of potentially contaminated stormwater and minimise uncontrolled erosion from occurring through catch drains and diversion channels; and
  - Early stabilization of diversion drains and bund. Stabilization methods include (i) meshing, hydro-mulching and application of suitable rapid germination species in drains, and (ii) use of constructed riprap and gabion rock baskets at critical sites such as drainage, confluences and outfalls into existing drainage sites.
- Water Protection
  - Wastewater discharges to water sources and effluent will be treated prior to any discharge; and
  - A formal groundwater monitoring program will be established as the mine enters operational phase to include TSF, process plant and long term soil stockpiles to ensure integrity of groundwater resources.
- Waste Rock Drainage (WRD)
  - PAF material excavated from the mine during initial construction will be used in the TSF embankment. Any additional PAF material encountered after construction of the TSF embankment will be dumped directly into the TSF storage and to be immediately submerged under 4 m of water;
  - NAF waste excavated during initial construction will be used in construction of the plant site pad or used as slope protection for the plant site run-off dam;
  - Waste spoil from the access decline located on top of the ridge above the mine is expected to be NAF and will be dumped into the TSF catchment area. The TSF will collect any sediment in the run-off from this waste dump; and
  - If the PAF waste material excavated from the mine workings cannot be disposed in the TSF, it will be necessary to encapsulate PAF material within a purpose built low permeability waste dump.

20.14 Flora and Fauna

Land clearance for the mine will result in vegetation loss, leading to fragmentation of vegetation communities and an increase in the prevalence of weed species. This will limit the regeneration of forest areas in the vicinity and impact upon the productivity of agricultural land and native timber forestry production. Increased accessibility can also potentially lead to further exploitation of timber and native timber forest products.

Fauna species will also be affected on a similar scale. Potential impacts include habitat fragmentation, loss of habitat, displacement and/or loss of individual fauna species. The mining operations will also lead to an increase risk of exposure to tailings and disturbance due to blasting for fauna species.

Aquatic fauna species will potentially be impacted through an increase in sediment loads and chemical contaminants entering the West Ban Phuc Creek and Chen Stream. The increased sediment loads can lead to physical alteration of aquatic habitats, river morphology (smothering of habitat, altered flow regimes and/or increased flooding frequency), changes in water quality and chemistry and in stream trophic structure (including fish and macro invertebrate food supplies).

20.14.1 Mitigation Measures

The following mitigation measures have been implemented to reduce the potential impacts that mining activities may have upon the flora and fauna within and adjacent to the project:

- Flora
  - Site specific survey will be undertaken to identify any listed plant species which may occur within the disturbed area;
  - Project infrastructure will be located away from areas of high conservation value and significant native timber forest production collection areas;
  - Induction training undertaken by all employees on the importance of conserving the biodiversity and conservation values; and
  - Re-vegetation trials will be established to determine optimum rehabilitation works for the project landforms.
- Fauna
  - Where practical, areas around the TSF will be fenced off to minimize wildlife access;
  - All machinery will meet industry noise standards to meet Vietnamese regulatory noise levels and will have noise reduction systems fitted to reduce the potential impacts of noise on fauna species within the project area;
  - Appropriate local speed limits will be set for mine vehicles on all haulage routes;
  - Controlled release of mine waters and maintain a vegetated buffer zone along streams and rivers; and
  - Sensitive placement of haulage roads and mine infrastructure to reduce fragmentation of fauna habitat.

20.14.2 Air Quality

Impacts to air quality will arise largely during the construction phase i.e. vegetation and topsoil removal, transporting and dumping of waste, haulage of concentrate, grading and road maintenance, movement of trucks, emissions from vehicles and power generating equipment. The stockpiling of topsoil, mineral ore, waste and ore crushing activities during the construction and operational phases can potentially generate dust.

Dust generation has been estimated to be at 1.25 tonnes over the duration of the mine construction (i.e. excavation work and transport). The operation of the mine will require blasting as part of the ore extraction process, and this is expected to generate between 3.8 tonnes and 5.4 tonnes per annum during the operation of the mine.

Exhaust emissions associated with vehicles machines and equipment are likely to be generated at low levels and highly dispersed in a rural setting both during construction and operational phases. Types of pollutants arising from the use of gasoline and diesel reportedly include carbon monoxide (CO), sulphur dioxide (SO_2) and hydrocarbons (HCs) emissions. Air emissions modelling indicated that the mine is likely to comply with the Vietnamese standards in relation to PM_{10} and Total Suspended Particulate (TSP).

20.14.2.1 Mitigation Measures

The following mitigation measures have been implemented to reduce the potential impacts that mining activities may have upon the air quality:

- Traffic Management
  - Impose vehicle speed limits to minimize generation of dust emissions as a result of vehicular movements; and
  - Avoid overloading of truck overloading, resulting in spillage during loading, transportation and unloading.
- Mine Design and Operational Considerations
  - Minimize areas exposed to wind erosion and prevent the spillage of ore and concentrate during loading and haulage activities;
  - Watering unsealed roads, disturbed surfaces and access roads to reduce dust emissions;
  - Rehabilitation of disturbed land to be undertaken as soon as practicable;
  - Minimize free fall height in stockpiling and locating stockpiles away from prevailing wind directions;
  - Providing water sprays at crusher and conveyor transfer points; and
  - Enclosure of air emission sources and treatment of emission sources by suitable technologies.

20.14.3 Noise

Sources of noise from the operation of the mine include the operation of the process plant comprising of jaw crushers, conveyer belt, screens and other associated machinery. Noise emission levels at Ban Phuc (45 – 51 dB(A)) and Ban Khoa (51 – 54 dB(A)) villages are generally within the accepted Vietnamese noise standards (60 dB(A) for LA_{eq} between 7 am and 6 pm and 55 dB(A) for LA_{eq} between 6 pm and 10 pm).

Based on arithmetic calculations, the average LA_{eq} 1 hr noise level for a bypassing mining vehicle through the villages is expected to be around 47 dB(A) and is below the Vietnamese noise standards. The ESIA report identified occasional noise exceedances during peak operating times.

20.14.3.1 Mitigation Measures

The following mitigation measures have been implemented to reduce the potential impacts that mining activities may have upon the noise levels in the vicinity:

- Equipment and Machinery
  - Machinery being fitted with acoustic enclosures, noise attenuating shields and mufflers;
  - Maintenance of exhaust mufflers; and
  - Implement buffer zones for local communities and provide early notification to communities when noise levels are anticipated to exceed regulatory standards.

20.14.4 Waste Generation

Solid waste generation from the construction works is approximately (i) 733,944 m^3 for associated excavation earthworks, and (ii) 79,000 m^3 for waste rocks, soil and blasting activities. An additional 26,850.2 tonnes (or 268,501.5 tonnes over the lifespan of the nickel mine) of rocks and soil will be removed per annum during the operation of the mine.

Domestic waste generation has been estimated to be between 75 kg/day – 150 kg/day, with approximately 250 employees working on the project site during the peak of the development.

20.14.4.1 Mitigation Measures

The following mitigation measures have been implemented to reduce waste generation volumes during construction phase:

- Land Clearance Protocols
  - Clearance of vegetation will be restricted to the minimum required for the installation of the facilities.
- General Waste Management
  - Hazardous materials will be substituted by bio-degradable material or reused, where practicable;

   - Waste collected shall be segregated prior to treatment for recycling or final disposal; and
   - Disposal of wastes will be to engineered and approved facilities, and in accordance with established procedures and regulatory requirements.

20.14.5 Health and Economic

An increase in traffic movement through Ban Phuc and Ban Khoa villages may result in reduced amenity, potential injuries and/or deaths in these communities.

Whilst the BPNM Project will create employment opportunities to the local and surrounding residents, the inter-regional migration of people into the area seeking employment can lead to increase in social tensions, infrastructure (i.e. health facilities), social ills (i.e. crime, prostitution, drug use) and demands for land and water resources.

A number of health issues can also potentially arise from the development of the nickel mine. Types of possible health issues include increased levels of dust and noise, use of chemicals and fuels leading to land and water contamination and increase in communal spreading of infectious diseases.

20.14.5.1 Mitigation Measures

The following mitigation measures have been implemented to manage the potential socio-economic and health impacts:

- Corporate Social Responsibility
   - The AMR Recruitment Policy provides for preferential employment for local communities;
   - The Community Development Liaison Plan for BPNM Project is currently being prepared and will be completed before January 2013. The Community Development Liaison Plan provides protocols for engagements with the local communities; and
   - Financial donations for improvements in education and health facilities for local communities.

20.15 Mine Closure and Rehabilitation Program

The philosophies for the Mine Closure and Rehabilitation Program are (i) project area is to be used by local communities for subsistence farming, (ii) allow for sustainable use of land by the local communities, and (iii) return the land to its original conditions (consistent with surrounding physical and social environment with no ongoing maintenance required) within a reasonable timeframe.

The Mine Closure Plan (Reference No. 1859/QD-BTNMT) was approved by the Son La Province in October 2011. The following key conditions are stipulated within the regulatory approval:

- Drainage piping design is to take into consideration seepage from underground workings and treated to meet statutory standards before discharging into the environment;

- The TSF is to be reinforced with soil and rock, and tailings bench to be capped with a 0.5 m thick clay and flooded to prevent oxidation;

- Grading and compaction to be conducted for Waste Rock Dump, including placing a 0.5 m thick rich soil cover, building a drainage system and planting of acacia trees throughout the area;

- Dismantle works and equipment constructed and erected during the operational phase and management of environmental pollution;

- Conduct grading, planting of trees and improving the natural landscape within the industrial yard and auxiliary facilities; and

- Barbed wire fences to be erected and warning signs to be strategically placed around the mine.

20.15.1 Preliminary Mine Closure Plan (PMCP)

The development of the Preliminary Mine Closure Plan (PMCP) will be guided by the following principles:

- Where possible, rehabilitation will be undertaken progressively until the mine is closed so as to minimize the potential for land and water degradation and reduce visual impact;

- Vegetation rehabilitation will be undertaken using species endemic to the local area that are suitable to both the physiographic and the hydrographic features of landforms;

- Endemic species of value to the local community will be used on rehabilitated areas;

- The design of post closure landforms and the selection of revegetated species will be in accordance with the accepted principals of long-term sustainability;

- The stability of the newly prepared land forms (i.e. topsoiling) prior to the re-establishment of vegetation is to be protected via the construction of moisture retaining graded drains, water holding structures, and where appropriate, the use of cover crops to provide initial erosion prevention;

- Site specific trials and research will be conducted to ensure that rehabilitation plans are both appropriate and feasible for all mine landforms; and

- The TSF will be managed as a contaminated land site immediately after closure, but will eventually be incorporated into the long term land use currently existing in the area.

The development of detailed rehabilitation designs and implementation plans will be subjected to the results of rehabilitation trials will be carried out over the life of the mine. AMR will prepare a PMCP once the mining life of the deposit has been exhausted. The PMCP will be formulated in consultation with MONRE and DONRE. The following rehabilitation concepts are considered for the various mine structures:

20.15.2 Waste Rock Dumps

Waste rock dumps will be progressively rehabilitated over the life of the mine with the objective of creating a structurally stable, vegetated landform, suitable for low intensity agriculture and/or habitat.

The waste dump surfaces will be loped at an angle to control surface runoff with cut off drains and drop-structures installed around the WRDs during operations to direct runoff to surrounding settling basins. These structures are to be built to ensure limited contaminated runoff to local creeks.

20.15.3 Tailings Storage Facility

Given the size of the Ban Trang catchment area, the location of TSF and the likelihood of flash flooding occurring in the area during monsoonal periods, effective management of the TSF structure must be undertaken to ensure sustainable post mine land use development. This commitment will be in the form of a tailings management plan. Two options are presented for post tailings land use and will be further refined in consultation with Vietnamese Authorities as part of the final mine closure plan (FMCP), specifically in terms of the expected short-term and long-term uses of the site (i.e. tailings water cover and/or regenerated forest).

20.15.4 Permanent Water Cover

This particular tailings containment system is applicable for the BPNM Project as there is a positive water balance in the project area. Appropriate fencing, bunding and other protective measures ensuring public safety will need to be installed. In the future this permanent water cover land use may be turned into a functioning wetland or capped as below for use as regenerated riparian forest.

20.15.5 Capping

This process will involve the reshaping of the dam surface to divert runoff away from the final tailings surface. An inert material to prevent water movement out of the tailings is then placed on top of the tailings over which a free draining material is placed. The inert barrier material may be composed of the final tailings distributions from the process plant if the natural sediment build up from Ban Trang Creek is not sufficient for capping. Topsoil is then used to resurface the area, followed by revegetation using suitable plant species. Any surface drainage from the post closure tailings land use will be collected via suitably sized retention structures and/or constructed wetlands in order to provide time to maximize sediment retention and/or polishing of drainage.

20.15.6 Process Plant

The process plant, ROM pad and associated facilities will be dismantled and removed. Subject to stakeholder consultation, some infrastructure may be retained for hand-over following decommissioning works. The decommissioned area will be contoured, topsoiled and revegetated to enable low intensity agriculture and/or habitat to match the original land use.

20.15.7 Water Management Features

All water storage dams, silt traps and drains at the project site will be retained for future water supply and storage or be rehabilitated to constructed wetlands so as to polish site drainage.

20.15.8 Office Buildings, Administration and Ablutions Block

Subject to stakeholder consultation all of the offices, the mess and the ablutions block may either be removed and the impacted areas rehabilitated or they will be left in situ and handed over to stakeholders for post-closure operations.

20.15.9 Power

All electricity transmission infrastructures will be removed or handed over to the regional government at the completion of the closure program. All electricity generator sets will be removed from the site.

20.15.10 Roads

Roads constructed for the proposed mining operation will either be retained to enable site access during closure and during rehabilitation period or for local community use post closure. Where internal roads are not required post-closure they will be rehabilitated to agricultural and/or habitat land uses.

20.15.11 Exploration Areas

All open drill holes will be backfilled and tracks revegetated accordingly.

20.15.12 Seed and Plant Stock

Access to a large reserve and a wide variety of endemic seed stock is fundamental to the success of the rehabilitation program. It is proposed that a regional seed collecting program, employing local community members, would commence during the construction phase to allow sufficient time to commence rehabilitation trials and to vegetate diversion drains and bunds.

As part of rehabilitation program commitments, consideration will be given to assisting with the establishment of a nursery to supply plants to AMR for the rehabilitation program.

20.15.13 Mine Closure Maintenance and Monitoring

It is proposed that by the end of the first year of operation, detailed rehabilitation procedures and a maintenance schedule will be developed by AMR and incorporated into updated versions of the EMP and included in the FMCP. Rehabilitation plans will be updated annually and will cover:

- Seed collection protocols and seed germination and maintenance procedures;
- Soil moisture and compaction requirements prior to planting;
- Topsoil storage and maintenance;
- Seeding rates and scouring methods;
- Erosion control procedures, weed control and herbicide protocols;
- Access and fencing of germinating areas; and
- Criteria for assessing revegetation success, rehabilitation standards and completion status.

20.15.14 Rehabilitation Security Deposit

The Rehabilitation Security Deposit required under the approval is USD$ 240,000. The Rehabilitation Security Deposit was calculated in accordance with the Vietnamese legislation and approved (Approval No. 1859/QD-BTNMT) by the regulatory authorities in October 2011. To date, 6 lodgements (USD$ 144,000) have been made. The remaining five (5) lodgements for the Rehabilitation Security Deposit are to be lodged between 2013 and 2018.

20.15.15 Lease Relinquishment

On cessation of mining operations, the lease held by AMR will retain the project area until such time as Vietnamese relinquishment criteria are met. The final criteria for lease relinquishment are to be developed in consultation with the Vietnamese Government and are likely to include:

- Final landform stability;
- An agreed long term post mine land use for the TSF;
- Maintenance of downstream water quality by water quality monitoring programs;
- Establishment of self-sustaining vegetation on all rehabilitated areas, and
- Removal of all hardstand areas and infrastructure excluding those to be used post closure by local communities and governments.

20.16 Monitoring Programs

AMR currently conducts the following environmental monitoring (Table 67) during the construction phase:

Table 67 Environmental Monitoring Requirements

Monitoring Elements		Frequency
Climate and Methodology	-	Monthly
Surface Water Resources	Stream water depth	Monthly
	Stream flow velocity	Two (2) times
	Stream water quality	Three (3) times
Groundwater Resources	Groundwater levels	Monthly
	Groundwater quality	Monthly
Ambient Air Quality	-	Three (3) times
Air Emissions	-	-
Wastewater Discharge	-	-
Ambient Noise	-	Monthly
Vibration	-	Monthly

Overall, analytical monitoring results for air, water and noise parameters indicated compliance with the Vietnamese legislation. With the exception of occasional non-compliances with dust concentrations detected during January and February 2011, no other exceedances were observed. The high dust readings can be attributed to dry season weather, traffic conditions and repair works.

In 2012, AMR constructed a suspension bridge for pedestrian and motorcycle traffic over Soui Chen (or more commonly referred to as Chen Stream) at a cost of approximately USD $100,000. Completion of this project has enabled villagers from the far side of Chen Stream to gain better access to the communities and amenities along Highway Number 37.

AMR is currently in the process of developing an internal monitoring plan to help keep track of resettlement activities associated with the BPNM project.

21 Capital and Operating Costs

21.1 Capital Cost Estimates

Estimated capital expenditures for mine equipment costs and construction costs are based primarily on information provided by BPNM. Capitalised pre-production mining costs are based on the mine plan prepared by AMDAD. Expenditure is expressed in US Dollars and is un-escalated. The starting date for capital expenditure was 1 January 2013. The estimates are based on estimates supplied by management of the company and rates supplied by suppliers. The t:lb unit conversion is 2205.

21.1.1 Initial Capital Cost Estimate

Capital expenditure from 1 January 2013 to commencement of production in June 2013 totals approximately US$ 34.66M, inclusive of contingency of US$ 2.80M (Table 68).

Table 68 Initial Capital Cost Summary

Item	US$
Labour	6,538,390
Earthworks	4,419,106
Processing plant	9,446,431
Engineering	1,818,636
Commissioning	1,190,852
Mobile equipment and spares	2,753,602
Camp	308,073
HSE	65,600
Contingency (10%)	2,000,230
Total Construction	22,002,530
Mining equipment	500,000
Re Open Underground	200,000
Development	4,621,332
Contingency	798,200
Total Mining	6,119,532
Total Capital Cost	34,660,453

The project indices have been summarised in Table 69. The revenues royalties and tariffs have been summarised in Table 70 and operating costs, profits and capex have been summarised in Table 71.

Table 69 Project Indices

Description	Unit	Dec-13	Dec-14	Dec-15	Dec-16	Dec-17	Dec-18	LOM
Mine Production								
Development ore	t	82,658	141,814	131,976	90,738	72,418	-	519,603
Production ore	t	41,140	199,812	224,975	253,589	293,239	81,055	1,093,810
Total ore mined	t	123,798	341,626	356,951	344,327	365,657	81,055	1,613,413
Head grades								
Ni		1.2%	2.3%	2.2%	2.3%	2.2%	2.0%	2.2%
Cu		0.6%	1.1%	1.0%	1.0%	1.0%	0.9%	1.0%
Co		0.03%	0.1%	0.1%	0.1%	0.1%	0.0%	0.1%
Contained metal								
Ni	t	2,438	8,027	7,825	7,865	8,156	1,346	35,658
Cu	t	1,264	3,659	3,647	3,490	3,564	641	16,266
Co	t	57	200	185	182	195	26	845
Recoveries								
Recovery - Ni	t	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Recovery - Cu	t	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Recovery - Co	t	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Metal in concentrate								
Ni in Conc	t	2,073	6,823	6,651	6,685	6,933	1,144	30,310
Cu in Conc	t	1,201	3,476	3,465	3,316	3,386	609	15,452
Co in Conc	t	40	140	129	128	136	18	591
Concentrate tennor								
Ni in Conc		9.5%	9.5%	9.5%	9.5%	9.5%	9.5%	9.5%

Description	Unit	Dec-13	Dec-14	Dec-15	Dec-16	Dec-17	Dec-18	LOM
Cu in Conc		2.8%	4.9%	5.0%	4.7%	4.7%	1.8%	4.9%
Co in Conc		0.1%	0.2%	0.2%	0.2%	0.2%	0.1%	0.2%
Concentrate produced								
Dry concentrate	t	21,819	71,824	70,015	70,373	72,979	12,039	319,049
Wet concentrate	t	23,892	78,648	76,666	77,058	79,912	13,183	349,358

Table 70 Base Case Revenues, Royalties and Tariffs.

Description	Unit	Dec-13	Dec-14	Dec-15	Dec-16	Dec-17	Dec-18	LOM
Received Revenue								
Nickel	US$ 000s	28,981	95,402	92,998	93,473	96,935	15,991	423,780
Copper	US$ 000s	5,005	14,482	14,437	13,816	14,108	2,539	64,386
Cobalt	US$ 000s	377	1,331	1,232	1,215	1,297	173	5,626
Total received revenue	US$ 000s	**34,363**	**111,215**	**108,667**	**108,504**	**112,340**	**18,703**	**493,792**
Government Royalties and Tariffs								
Royalties								
Royalty rate		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	
Royalty cost	US$ 000s	3,436	11,121	10,867	10,850	11,234	1,870	49,379
Export tariff								
Tariff % - Ni, Co		20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
Tariff % - Cu		30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%
Tariff cost	US$ 000s	7,373	23,691	23,177	23,082	23,879	3,995	105,197

Table 71 Operating Costs, Profits and Capex

Description	Unit	Dec-13	Dec-14	Dec-15	Dec-16	Dec-17	Dec-18	LOM
Operating Costs								**US$/t ore**
Labour	US$ 000s	4,248	8,819	7,574	7,119	5,187	800	20.92
Mining cost	US$ 000s	4,150	11,760	11,241	9,962	9,954	1,780	30.28
Processing cost	US$ 000s	1,975	5 451	5 696	5 494	5 835	1 293	15.96
Environmental cost	US$ 000s	248	683	714	689	731	162	2.00
G&A expenses	US$ 000s	579	1,599	1 671	1 612	1 712	379	4.68
Contingency environmental, G&A and labor	US$ 000s	508	1,110	996	942	763	134	10%
Total production cost (excl tariff and royalty)	US$ 000s	14,100	37,295	35,566	33,531	32,180	5,868	**158,539**
Offsite costs	US$ 000s	1,672	5,505	5,367	5,394	5,594	,923	91.0
Total production cost (incl tariff and royalty)	US$ 000s	24,909	72,107	69609	67,464	67,293	,11 733	**313,116**
Profit								
Operating Profit	US$ 000s	9,454	39,108	39,057	41,041	45,047	6,970	**180,676**
Capex								
Total capex (including sustaining capex)	US$ 000s	37,554	6,705	2,956	1,078	585	17	**50,770**

21.1.2 Process Plant Construction Cost Estimate

Equipment and facility costs have been obtained from the following sources: -

- Construction expenditure – BPNM (see Table 72).
- Owner's management team and consultants – BPNM October 2012 Budget.
- TSF – BPNM 2012 budget (Knight Piésold quantities).

Table 72 BPNM Plant Capital Estimate

Area	Total (US$)
Area 100 - Crushing	69,036
Area 200 - Grinding	1,221,636
Area 300 - Flotation	156,285
Area 400 - Concentrate Dewatering	372,409
Area 500 - Tails Thickening	1,236,312
Area 600 - Reagents	258,556
Area 700 - Services	286,210
Area 800 - Plant Piping	1,400,836
Area 900 - Power and Reticulation	3,382,082
Area 1000 - Buildings	687,391
Area 1050 - Laboratory	375,678
Total	**9,446,431**

21.1.3 Tailing storage facility and runoff dam

The tailings storage facility (TSF) and plant site runoff dam construction are underway and due for completion before the wet season that generally occurs from July onwards.

Table 73. TFS Capital Estimate

Description	Amount US$
Section 1 - preliminary and general	**230,000**
Section 2 - tailings storage facility (stage 1)	
Section 2.1 - site preparation	36,000
Section 2.2 - embankment construction	2,778,610
Section 2.3 - runoff / seepage collection pond embankment	-
Section 2.4 - spillway	1,507,450
Section 2.5 - seepage weir	3,700
Section 2.6 –instrumentation	37,300
Section 2 - tailings storage facility (stage 1)	**4,363,060**
Section 3.1 - site preparation	7,100

Description	Amount US$
Section 3.2 - embankment construction	110,200
Section 3.3 - return water pump	37,500
Section 3.4 - spillway	46,700
Section 3.5 - instrumentation	1,200
Section 3 - plant site runoff dam	**202,700**

Note, TSF capital expenditure breakdown reflects total project budget

21.1.4 Consumable Spares – Maintenance, Liquids, Reagents.

The working capital adjustment is in addition to the working capital components included in mining and processing expenditure incurred during the pre-development stage. This includes consumables, reagents, maintenance spares, stockpile and first fills. These have been not been identified separately.

21.2 Operating Cost Estimates

21.2.1 Mining Cost Estimate

AMDAD prepared a mining cost estimate corresponding to the life-of-mine schedule based on the following unit costs and factors supplied by BPNM (Table 74).

Table 74 Unit cost parameters

Cost Item	Unit	Unit cost US$
Development - decline	$/m	2,304
Development – lateral access	$/m	1,841
Development - lateral other	$/m	1,841
Development – orebody sills Based on $1670.08/m for a 4m x 4m drive	$/t	34.10
Vertical development – 2m x 2m, or 1.8m dia	$/m	1,290
Vertical development – 3m x 3m or 2.4m dia	$/m	2,186
Production drilling	$/m	8.53
Production blasting	$/m	4.02
Production loading (conventional)	$/t	5.73
Production loading (remote)	$/t	7.74
Trucking - waste	$/t.km	2.87
Trucking - ore	$/t.km	2.87
Truck loading	$/t	0.55
Cable bolting	$/m	28.20
Contingency	%	15

Note, this cost estimate only accounts for mining costs based on unit rates provided to AMDAD. AMDAD understands that costs for Technical Services, Management, Supervision, Maintenance Overheads, and Employee Wages and Benefits are not included in this estimate. The COG model used assumes that $30.70/t ore is allocated *to these cost areas. (Including contingency). (This excludes labour at a cost of $20.92/t and capitalised mining costs)*

The resultant annual mining costs from application of these unit costs and factors to the schedule are summarized in Table 75.

Table 75 Summary of mining cost estimate, including unit cost contribution per tonne of ore

		$/t	Total	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Development									
	Decline	3.99	**6,429,958**	2,827,566	2,769,163	357,822	358,062	117,345	0
	Lateral waste - access	1.25	**2,016,778**	875,177	607,603	295,782	118,967	119,249	0
	Lateral waste - other	1.66	**2,685,272**	1,223,057	820,587	343,117	179,361	119,150	0
	Lateral orebody development	10.98	**17,717,126**	2,818,428	4,835,484	4,500,037	3,093,914	2,469,263	0
	Vertical development, op	0.19	**308,193**	308,193	0	0	0	0	0
	Vertical development, far	0.09	**143,077**	0	62,510	20,195	20,564	39,808	0
	Vertical development, rar	0.51	**815,692**	571,118	142,493	0	34,342	67,739	0
	trucking - waste	0.92	**1,484,579**	634,403	513,032	148,976	116,903	71,265	0
	trucking - ore	1.16	**1,871,405**	280,521	454,264	452,404	337,682	346,535	0
	trucking	0.32	**522,045**	152,111	163,874	93,664	63,887	48,508	0
Production									
	drilling	1.06	**1,710,312**	88,814	314,652	355,061	401,554	461,582	88,650
	blasting	0.36	**586,394**	22,055	107,120	120,610	135,950	157,206	43,454
	loading - conv	5.44	**8,779,136**	330,196	1,603,730	1,805,692	2,035,358	2,353,598	650,563
	loading - remote	3.15	**5,079,357**	191,042	927,873	1,044,722	1,177,600	1,361,724	376,397
	trucking - ore	2.39	**3,857,798**	129,720	660,443	771,818	913,541	1,038,454	343,822
	trucking	0.37	**602,963**	22,678	110,146	124,017	139,791	161,648	44,682
Services									
	Cablesbolts - access	0.07	**108,433**	48,020	40,123	10,191	6,440	3,660	0
	Cablesbolts - orebody	0.85	**1,368,306**	224,312	367,139	344,382	248,671	183,802	0
Contingency		5.21	**8,413,024**	8,413,024	1,612,111	2,175,035	1,618,273	1,407,388	1,368,081
TOTAL	**Total Mining Cost**	**39.98**	**64,499,848**	**12,359,521**	**16,675,271**	**12,406,762**	**10,789,974**	**10,488,618**	**1,779,702**

21.2.2 Process Plant Operating Cost Estimate

Details the processing costs over the mine life are outlined in Table 76.

Table 76 Processing Cost Summary

Area	US$
Process costs - life of mine (incl 10% contingency)	25,774,035
Owners administration - life of mine (incl 10% contingency)	11,856,722
Total	**37,600,756**

The annual process costs have been obtained from the following sources:

- Process expenditure comprises a mix of fixed costs and variable costs based on production levels and unit rates for each expenditure category (Table 77); and
- Owner's administration includes local executive management, overheads, and environmental costs.

BPNM unit processing costs estimates are summarised in (Table 77).

Table 77 Summary of BPNM Process Cost Estimate

Concentrate Processing Costs		*Management assumptions*
Major consumables & reagents		
Ball grinding media	US$/t ore	1.70
Crusher liners & screen cloths	US$/t ore	0.21
Ball mill liners	US$/t ore	1.22
Soda ash	US$/t ore	0.95
Collector - SEX	US$/t ore	0.11
Depressant - SMBS	US$/t ore	0.25
Depressant - CMC	US$/t ore	0.51
Frother IF50	US$/t ore	0.24
Flocculant	US$/t ore	0.39
Power		
Crushing and stockpile	US$/t ore	0.64
Grinding	US$/t ore	2.24
Floatation	US$/t ore	0.61
Concentrate dewatering	US$/t ore	0.21
Tailings	US$/t ore	0.47
Reagents storage & mixing	US$/t ore	0.09
Services	US$/t ore	0.40
Administration	US$/t ore	0.21
Maintenance & Services		
Minor spares & consumables	US$/t ore	2.78
Mine & concentrate samples	US$/t ore	1.28

Concentrate Processing Costs		*Management assumptions*
Sub-total processing cost	US$/t ore	14.51
Processing Contingency	%	10%
Contingency	US$/t ore	1.45
Total Processing cost	**US$/t ore**	**15.96**

21.2.3 Labour Operating Costs

The average life of mine labour costs is $20.92/t. This labour costs includes all labour associated with the project, including construction, mining, maintenance, processing and administration areas.

21.2.4 Offsite Costs

Offsite costs are summarized in Table 78.

Table 78 Offsite costs

Offsite costs		
Conc Grade control	US$/t wet conc	2.0
Sea freight & customs	US$/t wet conc	19.0
Cost of transport	US$/t wet conc	70.0
Contingency	%	10

22 Economic Analysis

The analysis is based on information prepared by BPNM.

The following assumptions where made:

- Mining is targeted to commence in March 2013 and finishes in April 2018. Whilst this technical report considers mining will be undertaken by an owner-mining team, AMR will continue to review the relative benefits of alternative approaches, including contractor mining.
- The capital expenditure is from 1 January 2013 (all prior capital expenditure is regarded as sunk capital)
- Both NPV and IRR are relatively high due to the fact that some seventy to eighty million of historic expenditures (capital and operating) have already been “sunk” into the project
- The project payback period is approximately fourteen months
- Project NPV and IRR are represented on a 100% basis and are based on unlevered cash flows
- Base case, high and low price options were based on the London Metal Exchange’s (LME) last twenty four month of historical prices.

22.1 Taxes, Royalties, Tariffs and Sales Agreement

The Vietnamese taxation system includes the following taxes:

- Corporate income tax (**“CIT”**);
- Value added tax (**“VAT”**);
- Special sales taxes;
- Withholding tax;
- Import and export tax;
- Technology transfer tax;
- Foreign contractor tax;
- Personal income tax; and
- Royalties

The most relevant and important taxes affecting the mining sector and BPNM in particular are outlined below:

22.1.1 Corporate Income Tax

A tax rate of 25% was used in the financial model and BPNM provided a supporting document.

22.1.2 Value Added Tax

BPNM has to import most of its process plant and mining equipment from outside Vietnam and under current regulations BPNM is exempt from import duty and VAT on these imports. It is noted that approximately 80% of the project plant and equipment have been delivered to site.

22.1.3 Royalties

A royalty rate of 10% was used in the financial model.

22.1.4 Export Tariffs

An export tariff of 20% for Nickel and Cobalt and 30% for Copper was used in the financial model.

22.1.5 Sales Agreement

On 28th April 2008, BPNM entered into an offtake agreement with Jinchuan Group Ltd. (**"Jinchuan"**) who agreed to purchase all nickel concentrates produced during the life of the initial Ban Phuc Project. The agreement also granted Jinchuan first refusal option on additional nickel concentrates that BPNM may produce from new projects other than Ban Phuc.

22.1.6 Project design criteria and upside potential

The target mining rate is limited by the orebody size and geometry and has been set at 1,000 tpd of massive sulphide ore. The process plant throughput has been designed to match the ball mill which is expected to have an annual capacity of 450,000 t. As the mine cannot match that, the plant will operate on a campaign basis until such time that additional ore sources, some of which have already been identified, can be exploited.

The metal distribution in the upper sections of the ore body lends itself to extraction of high grade zones early in the life of the project which will facilitate an early payback on capital. Annual concentrate production is expected to average around 70,000t and contain 6,400t of nickel, 3,200t of copper and 130t of cobalt.

The Mineral Reserve currently total 1.61Mt at 2.2% nickel, 1.0% copper and 0.05% cobalt giving a mine life of over 5 years. The disseminated resource offers potential upside for a larger, bulk mining operation. Another alternative is to mine a portion of these which are readily accessible from the underground infrastructure associated with the MSV and could, subject to metal prices, provide an excellent opportunity to extend the life of the project.

AMR has identified multiple opportunities to grow the project, including:

- Blending of selected higher-grade portions of the disseminated sulphide deposit located adjacent to the MSV and accessible from the planned underground infrastructure;
- Extension of the MSV at depth; and
- Processing of identified satellite deposits which are located within easy trucking distance from the Ban Phuc processing complex.

The project economic model was based on the mine plan and capital and operating cost estimates prepared by AMDAD and BPNM. The key assumptions are summarized in Table 79.

Table 79 Key Project Assumptions

Key Dates	**Units**	
Production Commences		30 June 2013
Production		
Ore mined	t	1,613,413
Ore treated	t	1,613,413
Ore grades (mine life average)		
Nickel	%	2.21
Copper	%	1.01
Cobalt	%	0.05
Process Recoveries		
Nickel	%	85
Copper	%	95
Cobalt	%	70
Concentrate Grades		
Nickel	%	9.50
Copper	%	4.86
Cobalt	%	0.19
Metal produced in concentrate		
Nickel	t	30,310
Copper	t	15,452
Cobalt	t	591
Tax and tariff rates		
Royalty	%	10
Corporate Income Tax Rate	%	25
Export Tariff Nickel and Cobalt	%	20
Export Tariff Copper	%	30
Constants		
Tonnes to Pounds		2, 204.62

A cash flow model has been prepared on a calendar year (January to December) basis. Net present values and Internal Rate of Return (IRR) are calculated at mid-year assuming cash flows are incurred evenly throughout the year.

The model includes owners' G&A expenditure directly attributable to the Project and mining exploration and includes allocated overheads. Taxation expense is calculated on Vietnam corporate income tax rates and applicable exemptions. The financial model results are summarized in Table 80.

Table 80 Key Financial Results

Item	Unit	Total
Metal Price (Base Case)		
Net Present Value (NPV) @ 12%	C 000s	65,929
Internal Rate of Return (IRR)	%	69
Metal Price (High Case)		
NPV @12%	US$ 000s	153,064
IRR	%	143
Metal Price (Low Case)		
NPV @12%	US$ 000s	19,859
IRR	%	29

The NPV and IRR are relatively high due to the fact that significant historical expenditure has already been "sunk" into the project

22.1.7 Sensitivity to Revenue Changes and Sensitivity Analyses

Three scenarios have been modelled:

- a base case;
- a high case; and
- a low case.

The metal input prices have been included in Table 81, Table 82 and Table 83. As can be seen in Figure 83, Figure 84 and Figure 85, the project is neither particularly sensitive to price, capex, opex, royalties, export tariff, or head grades. This is mainly due to the following environment:

- Significant sunk capital already "sunk" into the project; and
- Short rate of payback (about fourteen months).

The model would generate a negative NPV if the nickel price drops below the US$ 13,500/t barrier (assuming a copper price of US$ 6,724/t). The cobalt price does not have a significant impact on the model.

22.1.7.1 Base case

The base case analysis is based on average London Metal Exchange (LME) spot metal prices over a 24 month period as at 23 January 2013, run flat for the life of the mine

Table 81 Base case metal prices

Metal Prices	Units	Price
Nickel	US$/t	19,974
Copper	US$/t	8,333
Cobalt	US$/t	31,724

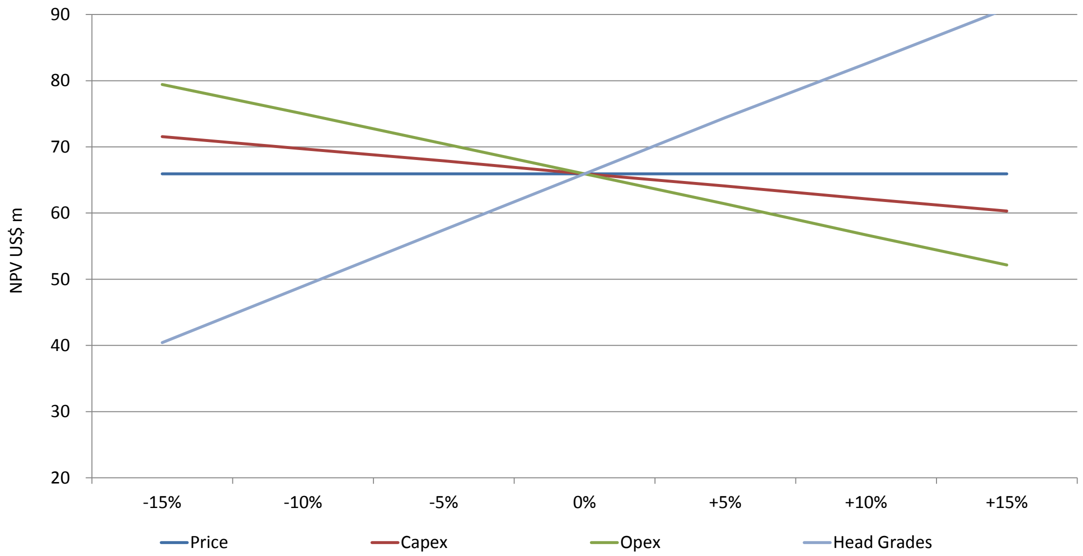


Figure 83 Base case sensitivity analysis

22.1.7.2 High case

The high case analysis is based on maximum spot metal prices over a 24 month period as at 23 January 2013, run flat for the life of the mine (Table 82).

Table 82 High case metal prices

Metal Prices	Units	Price
Nickel	US$/t	29,277
Copper	US$/t	10,185
Cobalt	US$/t	40,345

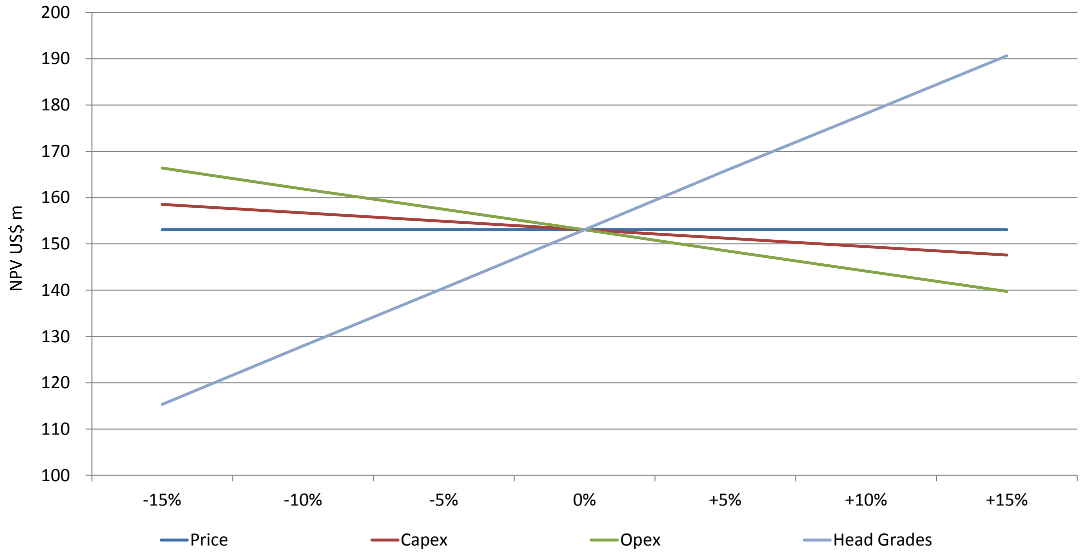


Figure 84 High case sensitivity analysis

22.1.7.3 The low case analysis

The low case analysis is based on minimum spot metal prices over a 24 month period as at 23 January 2013, run flat for the life of the mine (Table 83).

Table 83 Low case metal prices

Metal Prices	Units	Price
Nickel	US$/t	15,190
Copper	US$/t	6,724
Cobalt	US$/t	22,399

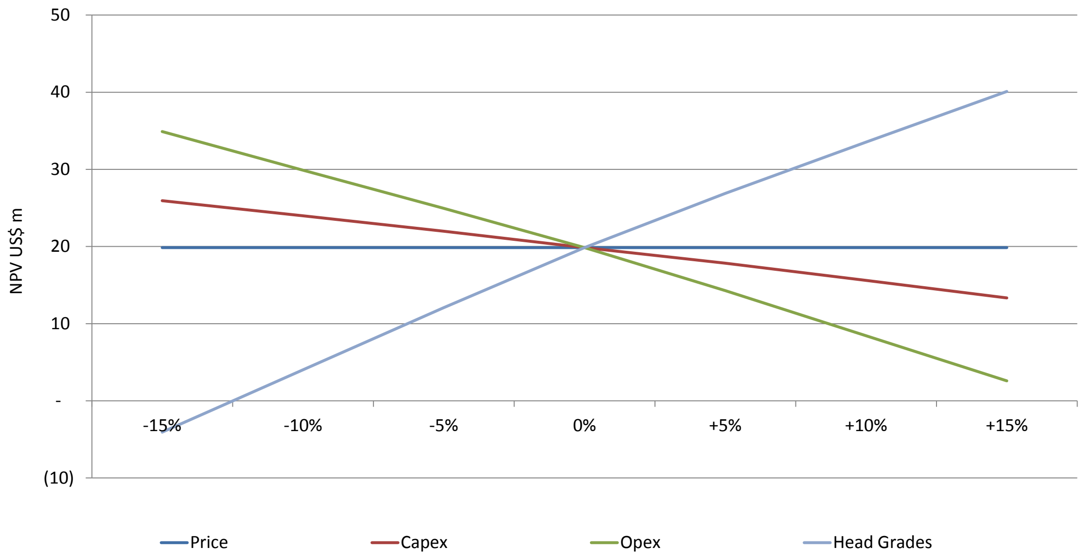


Figure 85 Low case sensitivity analysis

22.1.8 Payback

Under the metal prices assumed in the financial model base case, the payback from commencement of production is about fourteen months.

22.1.9 Mine Life

The mine life is just over five years.

22.1.10 Discounted cash flow

The discounted cash flow has been summarised in Table 78

Table 84 Discounted Cash flow

Free Cash Flow Analysis	Dec-13	Dec-14	Dec-15	Dec-16	Dec-17	Dec-18	LOM
EBITDA	8,563	38,003	37,952	39,936	43,942	6,602	174,997
Capital expenditures	-37,544	-6,705	-2,956	-1,078	-585	-17	-48,885
Changes in working capital	-10,716	1,850	257	-34	-103	8,747	0
Taxes	0	-1,449	-4,315	-4,994	-5,686	-522	-16,967
Project Post-tax unlevered free cash flow	-39,698	31,698	30,938	33,829	37,568	14,810	109,145

23 Adjacent Properties

This section is not applicable to the current report.

24 Other Relevant Data and Information

This section is not applicable to the current report.

25 Interpretation and Conclusions

25.1 Results and Interpretations

25.1.1 Mineral Resource

The validity of the database used for Mineral Resources estimate of mineralization at the Ban Phuc Nickel deposit has been confirmed via checks for internal consistency and accuracy. As a result of these checks the author considers that the drill hole data has been adequately validated and is appropriate for use in the estimation of an above the Mineral Resource estimated in this report

As a part of AMR's implemented field procedures, historical resource definition drilling at the Ban Phuc Nickel deposit were re-logged in order to upgrade the quality of the drillhole information and to improve the data collection process. The oxide boundary was re-digitised based on the new logging of weathering.

25.1.2 Metallurgical testwork

Marketable combined copper/nickel concentrates can be produced at high copper and nickel recoveries from the Massive Sulphide Vein at all locations within the deposit.

25.1.3 Mining Methods

The selected mine plan has been developed in close consultation with the geotechnical consultants to produce a simple mining method, whereby a number of areas of the orebody can be in production at one time, therefore providing a flexible mine to meet the needs of the production profile.

25.1.4 Environmental Studies

Overall, analytical monitoring results for air, water and noise parameters indicated compliance with the Vietnamese legislation. With the exception of occasional non-compliances with dust concentrations detected during January and February 2011, with no material impact to the project, no other exceedances were observed.

25.2 Risks and Uncertainties

25.2.1 Flotation cells

CSA raises the issue that the limited number of flotation cells selected for rougher, scavenger, cleaner and cleaner scavenger flotation duties could result in short-circuiting, and a larger number of smaller cells would be a preferred option.

25.2.2 Economic analysis

Three main project risks have been identified and while mitigation measures have been taken, they remain as risks to the project

- Nickel price
- Reliability of the productivity rates
- Reliability of projected Process plant throughput

25.2.2.1 Nickel prices

The nickel price has moved from a low of $9,000/t in 2008 to high of $29,000/t in 2011. (Figure 86)

Figure 86 Nickel price (2008-20012)

Nickel prices are the biggest risk to the project (Figure 86). In June 2012 the LME Nickel price touched on $17,000/t and then dropped to a low of just over $15,000/t in mid-August 2012. The price then rose to just under $19,000 at the end of September 2012 and then fell back down to the current price of about $16,000/t.

The model would generate a negative NPV if the Nickel price drops below the $ 13,500/t barrier (assuming a copper price of $ 6,724/t). The Cobalt price does not have a significant impact on the model.

25.2.2.2 Reliability of projected ramp up and productivity rates

Whilst the relevant Qualified Persons (AMDAD) have reviewed the orebody and conditions of the underground and opined that a 1,000 tpd mining rate is reasonably achievable. Maintaining this rate is a potential risk to the economics of the project. However, AMR has previously trained a team of 35 local miners, and in 2008 undertook key areas of underground development, including 1,000m of capital development and a 230m raise, to a high standard.

25.2.2.3 Reliability of projected process plant throughput

In terms of overall project risk, the relative risk of not achieving the design throughput of 450,000tpa is significantly mitigated by the scheduled annual ore feed rate to the plant. Annual ore production is approximately 360,000tpa. It is thus clear that the processing plant has significant excess capacity.

25.2.2.4 Mining equipment overhaul

While this only a relatively small amount ($500,000) this equipment will be required to produce the full tonnage for the life of the mine. The estimated amount is only to restore the equipment and there is no supporting document to confirm that this equipment will be able to meet the life of mine requirement.

25.2.2.5 Transport contracts:

BPNM is in the process of negotiating transport contracts.

25.2.2.6 Laboratory contract:

The decision as to contract this work internally or externally is still outstanding. While this is a relatively low risk, it is a contract that is required for a June 2013 start up.

25.2.2.7 Mining licence throughput:

BPNM are in the process of applying to have the annual throughput described in its increased from the current 200,000 tpa to 360,000 tpa, as required to support the business model.

26 Recommendations

26.1 Mineral Resource

Further drilling will need to be carried out if the remaining Inferred Mineral Resource estimate is to be upgraded to the Indicated status. The drilling program design should meet the following guidelines as a minimum:

- At a section spacing of 50m or less;
- On each 50m section, two drillholes giving intercepts that cover the full width of the MSV below the base of complete oxidation (BOCO); and
- Additional drillholes should aim to confirm the extrapolated depth extensions of the mineralized lithology (Table 85). The holes should also aim to delineate the orientation of the interpreted steeply structure. The drillhole intercepts for this purpose should cover mineralization projected below the present Mineral Resource base depth of 1200mRL down to 1100mRL.

Table 85 Recommended drilling and estimated costs

Type	Meters	Estimated cost US$
Drilling meters	3, 000	360,000- 380,000

The following suggestion would improve the confidence level and reliability of the resource estimate:

- Increase the number of density measurements; this should include measurement of core samples and downhole density logging.

The estimated cost to conduct this work would be from $5,000-$8,000.

26.2 Metallurgy

If it is decided to pursue the possibility of producing separate copper and nickel composites from the MSV then the following should be completed:

- Optimisation of flotation conditions on a sample with more representative nickel and copper head grades;
- Concentrate marketing studies. ;
- Revision of the plant flowsheet including management of the water circuits which may be problematic; and
- Revision of plant capital and operating costs.

The total estimated cost would be $80,000-$120,000.

26.3 Economic Analysis

CSA recommends the following that AMR consider the following:

- Recruit and commence training of the underground team as quickly as possible;
- Obtain written quotes from the original equipment manufacturers for the underground equipment (this equipment is expected to be used for the life of the mine);
- Obtain signed contracts for the transport of concentrate;
- Obtain signed contracts for the laboratory contract;
- Complete the ongoing Mining Licence amendment process; and
- Studies should be carried out to assess the options for increasing the life of the mine.

A budget of $100,000 - $150,000 should be allocated to the above recommendations.

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Glossary of Technical Terms and Abbreviations

Abbreviation	Description
%	Per cent
3D	Three-dimensional model or data
AMMTEC	AMMTEC Laboratories
AMR	Asian Mineral Resources Limited
ANC	acid neutralizing capacity
ARD	acid rock drainage
Ag	Silver
ASCII	Digital computer code containing text data
Au	Gold
azimuth	Drill hole azimuth deviation (from north)
binary	Digital file containing characteristics readable by computer only
ºC	Celsius degrees
clipping window	In case of display of three-dimensional data at the plane, plus minus the distance, within which the data is projected perpendicular to the image plane
cm	centimetre
coefficient of correlations	Statistical measure of the degree of similarity between two parameters
coefficient of variation (CV)	In statistics, the normalized variation value in a sample population
collar	Geographical coordinates of the collar of a drill hole or a working portal
compositing	In sampling and resource estimation, process designed to carry all samples to certain equal length
core sampling	In exploration, a sampling method of obtaining ore or rock samples from a drill hole core for further assay
CSA	CSA Global Pty Ltd
CSV	Digital computer file containing comma-separated text data
cut -off grade	The threshold value in exploration and geological resources estimation above which ore material is selectively processed or estimated
d	Diameter
declustering	In geostatistics, a procedure allowing bounded grouping of samples within the octant sectors of a search ellipse
digital terrain model	Three-dimensional wireframe surface model, for example, topography (DTM)
DIP	Angle of drilling of a drill hole
expedition	In CIS countries, a state territorial exploration enterprise
flagging	Coding of cells of the digital model
FROM	Beginning of intersection

g	gram
geochemical sampling	In exploration, the main method of sampling for determination of presence of mineralization. A geochemical sample usually unites fragments of rock chipped with a hammer from drill hole core at a specific interval
geometric mean	The antilog of the mean value of the logarithms of individual values. For a logarithmic distribution, the geometric mean is equal to the median. For a logarithmic distribution, the geometric mean is equal to the median
group sampling	In exploration and mining, method of sampling by means of union of the material of individual samples characterizing an independent orebody
histogram	Diagrammatic representation of data distribution by calculating frequency of occurrence
JORC Code	Australasian Joint Ore Resources Committee Code
kg	kilogram
km	kilometre
Kriging	Method of interpolating grade using variogram parameters associated with the samples' spatial distribution. Kriging estimates grades in untested areas (blocks) such that the variogram parameters are used for optimum weighting of known grades. Kriging weights known grades such that variation of the estimation is minimised, and the standard deviation is equal to zero (based on the model)
lag	The chosen spacing for constructing a variogram
lognormal	Relates to the distribution of a variable value, where the logarithm of this variable is a normal distribution
m	meter
M	million or mega (106)
macro	A set of MICROMINE commands written as a computer program for reading and handling data
mean	Arithmetic mean
median	Sample occupying the middle position in a database
Micromine Consulting	Consulting division of Micromine Pty Ltd
MICROMINE	Software product for exploration and the mining industry
ml	millilitre
ml/l	millilitre per litre
mm	millimetre
MSV	Massive sulphide veins
Mt	million tonnes
NI 43-101	National Instrument 43-101
nugget effect	Measure of the variability during repeat analysis of a sample due to a measurement error or the presence of natural, small-scale variability. Although the variogram value at 0 spacing should be equal to zero, these factors may affect the values of samples taken at a very short distance from each other such that their values may vary. A vertical jump from the zero value at the origin of a variogram with very small spacing is called the nugget effect.
omni	In all directions
overburden	All material above mineralization

percentile	In statistics, one one-hundredth of the data. It is generally used to break a database down into equal hundredths
population	In geostatistics, a population formed from grades having identical or similar geostatistical characteristics. Ideally, one given population is characterized by a linear distribution
probability curve	Diagram showing cumulative frequency as a function of interval size on a logarithmic scale
quantile plot	Diagrammatic representation of the distribution of two variables. It is one of the control tools, e.g., when comparing grades of a model with sampling data. It is one of the control tools, e.g., for comparing model grades with sampling data
quantile	In statistics, a discrete value of a variable for the purposes of comparing two populations after they have been sorted in ascending order.
range	Same as Influence Zone; as the spacing between pairs increases, the value of corresponding variogram as a whole also increases. However, the value of the mean square difference between pairs of values does not change from the defined spacing value, and the variogram reaches its plateau. The horizontal spacing at which a variogram reaches its plateau is called the range. Above this spacing there is no correlation between samples.
reserves	Mineable geological resources
RL	Elevation above the sea level
RL	Elevation of the collar of a drill hole, a trench or a pit bench above the sea level
run m	run meter
sample	Specimen with analytically determined grade values for the components being studied
scatter plot	Diagrammatic representation of measurement pairs about an orthogonal axis
SG	Specific Gravity
sill	Variation value at which a variogram reaches a plateau
standard deviation	Statistical value of data dispersion around the mean value
string	Series of 3D points connected in series by straight lines
t	ton
t/m3	ton per cubic meter
team (party)	In CIS countries, a state specialized geological enterprise, usually a part of an expedition (see expedition)
TO	end of intersection
unfolding	Computer program function allowing data of folded structures to be unfolded onto a plane using control frames and strings
variation	In statistics, the measure of dispersion around the mean value of a data set
variogram	Graph showing variability of an element by increasing spacing between samples
variography	The process of constructing a variogram
wireframe model	3D surface defined by triangles
X	Coordinate of the longitude of a drill hole, a trench collar, or a pit bench
Y	coordinate of the latitude of a drill hole, a trench collar, or a pit bench
y	year