2 November 2017

# Application lodged for secondary listing on TSX

# Project renamed Clean TeQ Sunrise

Mr Robert Friedland and Mr Jiang Zhaobai, Co-Chairmen of Clean TeQ Holdings Limited (**Clean TeQ** or **Company**) (CLQ:ASX; CTEQF:OTCQX), and Mr Sam Riggall, Chief Executive Officer, today announced that the Company has lodged a formal application for a secondary listing of its ordinary shares on the Toronto Stock Exchange (**TSX**), Canada's preeminent stock exchange and one of the world's largest and most liquid exchanges.

The Company's ordinary shares will continue to trade under the symbol CLQ on the Australian Stock Exchange and CTEQF on the United States OTCQX Exchange.

Co Chairman Mr Robert Friedland noted, *"We are seeing a significant increase in the level of North American investor awareness and interest in Clean TeQ. A secondary listing on the TSX will provide improved accessibility and liquidity for a broad range of retail and institutional investors in Canada and the US."*

Recent feedback from investors in Canada and the US indicates very strong interest in the battery materials sector and in companies such as Clean TeQ, which offer direct exposure to world-class upstream assets and proprietary processing technologies. A dual-listing on the TSX is an outstanding opportunity to raise the Company's profile in the North American market and will enable a substantially larger pool of investors to invest in the Company.

In recognition of the growing global profile of Clean TeQ and its Syerston Project, the name of the Syerston Project will be changed to "Clean TeQ Sunrise" (**Project**). The new name signifies the change in focus of the Project as an emerging global source of cobalt sulphate, nickel sulphate and scandium, and provides a strong connection to the local area. Sunrise is the name of a property located to the south-west of the project area which is owned by Clean TeQ.

Co-Chairman Mr Jiang Zhaobai commented: *"We are delighted to be announcing the re-naming of our Project to the Clean TeQ Sunrise Project. The name not only recognises our connection to the local area, but also the significant progression in the life cycle of this Project under the ownership of Clean TeQ. Clean TeQ Sunrise is emerging as a key and highly strategic source of high-quality raw materials for the* *battery industry and all of us at Clean TeQ are excited about the value the Project can deliver to our stakeholders."*

The application is subject to review and approval by TSX. Subject to the application being approved, the listing process is expected to be completed before the end of 2017.

To support the TSX listing application the Company has made available on its website at www.cleanteq.com the attached National Instrument 43-101 (**NI 43-101**) technical report for the Clean TeQ Sunrise Project (**Report**) which has been prepared by SRK Consulting (Australia) Pty Ltd (**SRK**). While the Report is based on the 2016 Pre-Feasibility Study (**2016 PFS**), SRK has updated several assumptions including capital and operating cost estimates based on the latest estimates of project development and operating costs. The NI 43-101 Report has been lodged with TSX as part of the listing application.

Specifically, the Report estimates total capital costs for the Project to be US$784M (including a contingency of US$102 million) compared to the original 2016 PFS capital cost estimate of US$680M (including US$62 million contingency). SRK have also revised operating costs per pound of nickel produced (including cobalt credits) from the 2016 PFS assumption of US$0.89/lb to US$1.42/lb. The increases to capital and operating costs are due to re-estimation of the processing capital and operating costs based on latest designs as well as updated labor rates and raw material costs. While these changes have resulted in a minor reduction in the post-tax internal rate of return from 25% to 21%[1], the economics of the Project, at conservative commodity price forecasts (US$7.50/lb Ni and US$14.00/lb Co, inclusive of sulphate premia), remain strong.

The NI 43-101 is based on the 2016 PFS resource and mining schedule. A further update to the economics of the Project will be provided upon completion of the Definitive Feasibility Study (**DFS**). The DFS will incorporate an updated mine plan and production schedule which will be based on the 2017 Resource update. For full details of the 2017 Resource update please see the ASX announcement of 9 October 2017.

**For more information about Clean TeQ contact:**


| | |
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**About Clean TeQ Holdings Limited (ASX: CLQ)** – Based in Melbourne, Clean TeQ, using its proprietary Clean-iX® continuous ion exchange technology, is a leader in metals recovery and industrial water treatment.

For more information about Clean TeQ please visit the Company's website www.cleanteq.com.

[1] Post tax, 8% discount, 100% equity, real terms

**About the Clean TeQ Sunrise Project** – Clean TeQ is the 100% owner of the Clean TeQ Sunrise Project, located in New South Wales. The Clean TeQ Sunrise Project is one of the largest cobalt and nickel deposits in the developed world, and one of the largest and highest-grade accumulations of scandium ever discovered.

**About Clean TeQ Water** – Through its wholly owned subsidiary Clean TeQ Water, Clean TeQ is also providing innovative wastewater treatment solutions for removing hardness, desalination, nutrient removal, zero liquid discharge. The sectors of focus include municipal wastewater, surface water, industrial waste water and mining waste water.

For more information about Clean TeQ Water please visit www.cleanteqwater.com

*This release may contain forward-looking statements. The actual results could differ materially from a conclusion, forecast or projection in the forward-looking information. Certain material factors or assumptions were applied in drawing a conclusion or making a forecast or projection as reflected in the forward-looking information.*

# Syerston Nickel Cobalt Project, New South Wales, Australia NI 43-101 Technical Report

**Prepared for:**


## Clean TeQ Holdings Limited

**12/21 Howleys Road**

**Notting Hill, Victoria, 3168  Australia**


**Prepared by:**


## SRK Consulting (Australia) Pty Ltd

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**SRK Project Number:  CTQ001**

**Qualified Persons:**


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Peter Kitto, BSc (Geology)(Hons), FAusIMM, Principal Consultant, Development and Mining Services

Lynn Widenbar, BSc (Geology), MSc (Mineral Exploration) (Hons), DIC, MAusIMM, MAIG, Principal Consultant, Widenbar and Associates Pty Ltd

Simon Walsh, BSc (Extractive Metallurgy & Chemistry), MBA (Hons), MAusIMM(CP), GAICD, Associate Principal Consultant (Metallurgy)



**Effective Date / Date of Report:** **30 October 2017**

# Date and Signature Page

**SRK Project Number: CTQ001**

## SRK Consulting (Australasia) Pty Ltd

Level 1, 10 Richardson Street
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**Clean TeQ Holdings Limited**
**Syerston Nickel Cobalt Project**
**New South Wales, Australia**

**NI 43-101 Technical Report**

**Project Manager: Peter Fairfield**

**Date of Report: 30 October 2017**

**Effective Date: 30 October 2017**

**Signature Qualified Persons:**

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# Important Notice

SRK has prepared this Technical Report for Clean TeQ as a National Instrument 43-101 Technical Report, as prescribed in the Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101). The data, information, estimates, conclusions and recommendations contained herein, as prepared and presented by the Authors, are consistent with:

- information available at the time of preparation;
- data supplied by outside sources, which has been verified by the authors as applicable; and
- the assumptions, conditions and qualifications set forth in this Technical Report.

## NON-IFRS MEASURES

This Technical Report contains certain non-International Financial Reporting Standards (IFRS) measures. Such measures have non-standardised meaning under the IFRS and may not be comparable to similar measures used by other issuers.

### Table of Authors and Qualified Persons (QP)

| Section | Description | Nominated QP |
|---|---|---|
| 1 | Executive Summary | Peter Fairfield |
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| 3 | Reliance on Experts | Peter Fairfield |
| 4 | Property Description and Location | Peter Fairfield |
| 5 | Accessibility, Climate, Local Resources, Infrastructure and Physiography | Peter Fairfield |
| 6 | History | Danny Kentwell |
| 7 | Geological Setting and Mineralisation | Danny Kentwell |
| 7.2.6 | 2017 Mineral Resource Estimate Interpretation | Peter Kitto |
| 8 | Deposit Types | Danny Kentwell |
| 9 | Exploration | Danny Kentwell |
| 10 | Drilling | Peter Kitto |
| 11 | Sampling Preparation, Analyses and Security | Danny Kentwell |
| 11.8 | 2017 Estimate QA/QC Drill Hole Series Validation | Peter Kitto |
| 11.9 | I2017 Estimate Independent custody sampling | Peter Kitto |
| 12 | Data Verification | Danny Kentwell |
| 12.4 | 2017 Data Validation | Peter Kitto |
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| 14.1 | Mineral Resource Estimates | Danny Kentwell |
| 14.2 | Mineral Resource Estimate Update 2017 | Lynn Widenbar |
| 15 | Mineral Reserve Estimates | Peter Fairfield |
| 16 | Mining Methods | Peter Fairfield |
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| 18 | Project Infrastructure | Peter Fairfield |
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| 21 | Capital and Operating Costs | Peter Fairfield |
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# Disclaimer

The opinions expressed in this Report have been based on the information supplied to SRK Consulting (Australasia) Pty Ltd (SRK) by Clean TeQ Holdings Limited (Clean TeQ). The opinions in this Report are provided in response to a specific request from Clean TeQ to do so. SRK has exercised all due care in reviewing the supplied information. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data.

Opinions presented in this Report apply to the site conditions and features as they existed at the time of SRK's investigations, and those reasonably foreseeable. These opinions do not necessarily apply to conditions and features that may arise after the date of this Report, about which SRK had no prior knowledge nor had the opportunity to evaluate.

# List of Abbreviations

| Abbreviation | Meaning |
|---|---|
| 2D | two dimensional |
| 3D | three dimensional |
| AAS | atomic absorption spectroscopy |
| AGD | AGD Mining Pty Ltd |
| ALS | ALS Minerals |
| Amdel | Amdel Limited Mineral Services Laboratory |
| ANFO | ammonium nitrate-fuel oil |
| ASL | above sea level |
| ATCF | after tax cash flow |
| Ni | Nickel |
| NiEq | Nickel equivalent |
| AUD | Australian dollar |
| BAppSc | Bachelor of Applied Science |
| BCom | Bachelor of Commerce |
| BD | bulk density |
| BEng | Bachelor of Engineering |
| BSc | Bachelor of Science |
| CIM | Canadian Institute of Mining, Metallurgy and Petroleum |
| Clean TeQ | Clean TeQ Holdings Ltd |
| Co | Cobalt |
| CRF | cemented rock fill |
| dmt | dry metric tonne |
| DTM | digital terrain model |
| EM | electromagnetic |
| EPA | Environmental Protection Agency |
| EVCs | Ecological Vegetation Classes |
| FAR | fresh air rise |
| FAusIMM | Fellow of The Australasian Institute of Mining and Metallurgy |
| GDip | Graduate Diploma |
| GEF | Gold and Exploration Finance Company of Australia |
| GMA/WMC | Gold Mines of Australia/Western Mining Corporation |
| GPS | global positioning system |
| GST | goods and services tax |
| g/t | grams per tonne |
| HBr | hydrobromic acid |
| HCl | hydrochloric acid |
| HR | hydraulic radius |
| ICP - AES | inductively couple plasma atomic emission spectroscopy |
| $ID^2$ | inverse distance squared |

| Abbreviation | Meaning |
|---|---|
| $ID^3$ | inverse distance cubed |
| IRR | internal rate of return |
| kg | kilogram |
| kL | kilolitre |
| km | kilometre |
| Koz | kilo ounces |
| kt | kilotonne |
| ktpa | kilotonnes per annum |
| ktpm | kilotonnes per month |
| kV | kilovolt |
| kVA | kilovolt ampere |
| kW | kilowatt |
| kWh | kilowatt hour |
| L | litres |
| LOM | life of mine |
| L/s | litres per second |
| M | million |
| Ma | million years |
| MAusIMM(CP) | Chartered Professional Member of The Australasian Institute of Mining & Metallurgy |
| mg/kg | milligrams per kilogram |
| mg/L | milligrams per litre |
| mH | metres high |
| ML | million litres |
| mm | millimetres |
| MMI | mobile metal ion |
| Moz | million ounces |
| mRL | metres reduced level |
| Mtpa | million tonnes per annum |
| $m^3$ | cubic metres |
| $m^3$/s | cubic metre per second |
| $m^3$/s/KW | cubic metre per second per kilowatt |
| MVA | megawatt ampere |
| mW | metres wide |
| MW | megawatt |
| Ni | Nickel |
| NiEq | Nickel equivalent |
| NI 43-101 | National Instrument 43-101 |
| NPV | net present value |
| oz | ounces |
| PEA | Preliminary Economic Assessment |

| Abbreviation | Meaning |
|---|---|
| PhD | Doctor of Philosophy |
| QA/QC | quality assurance/quality control |
| QP | Qualified Person |
| RAR | return air raise |
| RC | reverse circulation |
| RO | reverse osmosis |
| ROM | run-of-mine |
| RPD | relative paired difference |
| SAA | recent shallow alluvial aquifer |
| SD | standard deviation |
| SRK | SRK Consulting (Australasia) Pty Ltd |
| t | tonnes |
| tpa | tonnes per annum |
| t/mth | tonnes per month |
| TSF | tailings storage facility |
| TSX | Toronto Stock Exchange |
| USD | United States dollar |
| V | volt |

# 1 Summary

## 1.1 Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) has prepared this Technical Report for Clean TeQ Holdings Ltd (Clean TeQ or the Company), who, through its wholly owned subsidiary, Scandium21 Pty Ltd (Scandium21), owns the Syerston Project (Project) in central New South Wales, Australia.

This Technical Report provides an analysis of mining and processing of ore through a high pressure acid leach (HPAL)/ resin-in-pulp (RIP) process, to determine the economics of producing hydrated nickel sulphate ($NiSO_4.6H_2O$) and hydrated cobalt sulphate ($CoSO_4.7H_2O$).

The Technical Report is based on a 2016 pre-feasibility study (PFS) entitled Syerston Mine Pre-Feasibility Study, Nickel & Cobalt and dated October 2016 (2016 PFS), which was prepared for Clean TeQ. SRK has reviewed and validated the basis of and results contained in the 2016 PFS report and supporting information and has made changes where appropriate.

Following completion of the 2016 PFS report, Clean TeQ released Mineral Resource and Ore Reserve estimates to the Australian Securities Exchange (ASX) under the guidelines of the JORC Code (2012 edition).

An updated Mineral Resource Estimate (2017 Resource) was produced by independent consultants Widenbar & Associates Pty Ltd (Widenbar), and Development & Mining Services (Kitto) in October 2017. This Mineral Resources was released to the Australian Securities Exchange (ASX) under the guidelines of the JORC Code (2012 edition) and is reported in this report

**The reported Mineral Reserve is based on the Mineral Resource prepared in the 2016 PFS and not the 2017 Resource.**

The key parameters of the Project are detailed in Table 1-1.

**Table 1-1: Syerston project – key parameters**

| Parameter | Assumption/ Output |
|---|---|
| Resource Base used for mine planning | 2016 Measured & Indicated Resource - JORC Code (2012) |
| Autoclave Throughput | 2.5 Mtpa |
| Average Strip Ratio | 0.8:1 |
| Autoclave Average Feed Grade (Years 3 -20) | |
| *Nickel* | 0.80% |
| *Cobalt* | 0.14% |
| Average Production (Years 3 - 10) | |
| *Nickel as Ni* | 21,172 tpa |
| *Cobalt as Co* | 3,919 tpa |
| Average Production (Years 3 - 20) | |
| *Nickel as Ni* | 18,730 tpa |
| *Cobalt as Co* | 3,222 tpa |
| Average Recoveries | |
| *Nickel* | 93.5% |
| *Cobalt* | 92.7% |

| Parameter | Assumption/ Output |
|---|---|
| Life of Mine | 39 years |
| Life of Mine for Financial Modelling | 20 years |
| Long-term Price Assumption | |
| *Nickel, based on $NiSO_4.6H_2O$ Product* | USD7.50/lb |
| *Cobalt, based on $CoSO_4.7H_2O$ Product* | USD14.00/lb |
| Foreign Exchange Rate | 0.75USD:1.00AUD |
| Total Capital Cost | |
| | AUD1,045M |
| | USD784M |
| Average C1 Cash Cost (Year 3 - 20) [1] | |
| *without Co by-product credits* | USD3.86/lb |
| *with Co by-product credits* | USD1.40/lb |
| Net Present Value ($NPV_8$) – post tax | USD747.5M |
| Internal Rate of Return (IRR) – post tax [2] | 21.0% |

Notes:

1 C1 cash cost does not consider any potential by-product revenue from scandium oxide sales, royalties, depreciation or sea freight of nickel and cobalt sulphate.

2 Post tax, 8% discount, 100% equity, real terms.

### 1.1.1 Study methodology

The deposit is significantly developed – two feasibility studies for 2.0 Mtpa (2000) and 2.5 Mtpa (2005) nickel and cobalt operations were completed by the previous owners, Black Range Minerals (Black Range) and Ivanplats Syerston (Ivanplats). In addition to these historical studies, the environmental impact statement (EIS) completed in 2000 was referenced for potential environmental impacts and methodologies for mitigation of these impacts.

The 2016 PFS reported a Mineral Resource in accordance with the JORC Code (2012). The Indicated and Inferred Mineral Resource was completed by McDonald Speijers Pty Ltd (McDonald Speijers), based on more than 1,200 historical drill holes. Results from the recent drilling on high grade scandium areas on the periphery of the deposit have not been included in this resource estimate. Based on technical and cost inputs from historical studies, Mine Pit Modelling and Production Scheduling for the 2005 Feasibility Study were used as the basis for the 2016 PFS. The 2916 Resource reporting has been updated under 2012 JORC Code guidelines; the original 2004 mine plan was adopted.

Due to the substantial amount of metallurgical testwork carried out by Black Range and Ivanplats for the Project, limited additional metallurgical testwork was carried out, with the exception of nickel and cobalt resin loading, which used historical composite samples. All other process design criteria for the resin sections was sourced from laterite development work completed by Clean TeQ between 2000 and 2008.

## 1.2 Project overview

### 1.2.1 Project location

The Syerston deposit is situated in central New South Wales, about 350 km WNW of Sydney. The Project is well supported by major centres, with the mining communities of Parkes, Dubbo and Condobolin all located within 100 km of the Project area. The local town is Fifield which is located 4 km from the Project area (Figure 1-1). The Project area experiences a subtropical dry climate, i.e. very low rainfall, high daytime temperatures in summer and low minimum temperatures in winter.

The district is predominantly used for agriculture, with crops in the region including wheat, barley and oats. Sheep and cattle grazing is also common throughout the district. Due to widespread clearing for agriculture over the last 100 years, very little of the original vegetation remains.

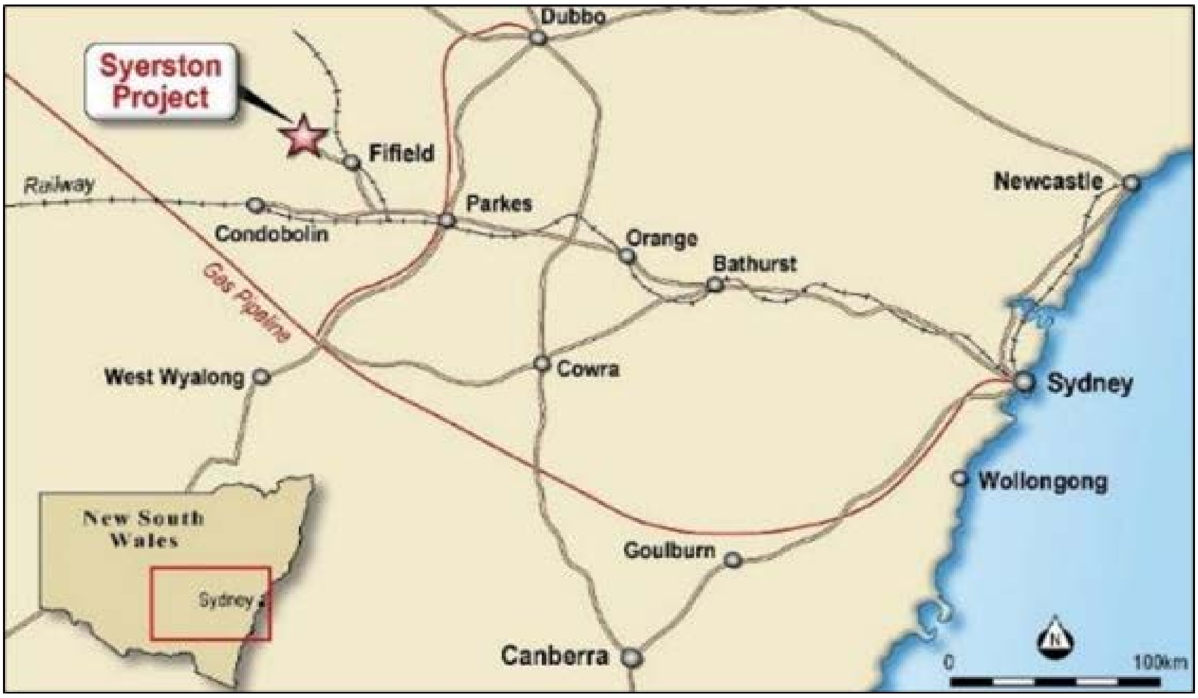



**Figure 1-1: Project location plan**

The Project area is located on three pastoral properties and includes previously mined land (magnesite), State Forest and Crown Land. The Fifield State Forest occupies a small part of the Project area situated along the northern border, and the Unoccupied Crown Land is found in the north-eastern corner of the Project area (Figure 1-2).

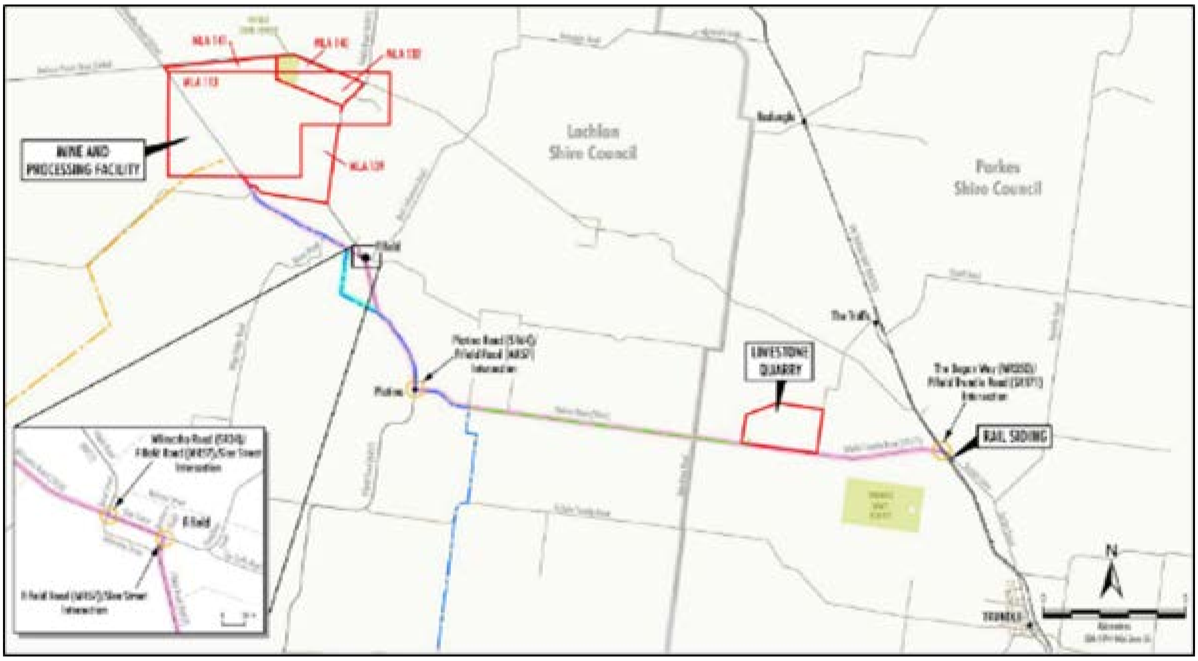



**Figure 1-2: Layout or proposed infrastructure and State Forests**

One of the Project's competitive advantages is its proximity to existing infrastructure. The Project is located ~80 km from to the Moomba–Sydney natural gas pipeline, a rail line within 20 km of the Project and bitumen roads providing good access to the site. The major centres have excellent infrastructure including transport, airport and rail facilities, all of which can service the Project's requirements. The Project and associated infrastructure are located within the shires of Lachlan and Parkes. The borefield providing the bulk of the water for the Project is located in the Forbes Shire.

### 1.2.2 Mineral titles and landholding

The Exploration Licence (EL) and Mining Lease Applications (MLAs) for the Project are shown in Figure 1-3.

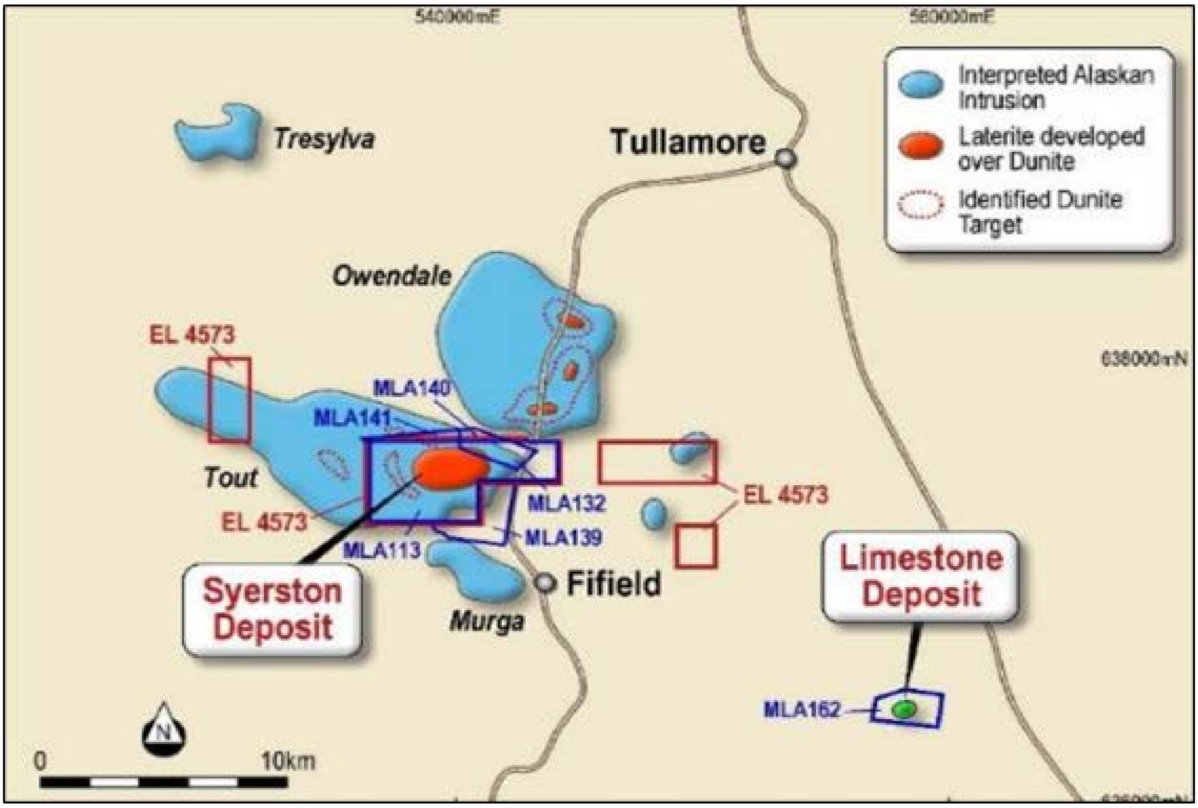



**Figure 1-3: Tenement ownership overview**

EL 4573 covers the area of the Project and its associated facilities. EL 4573 has been granted, subject to the provisions of the *Mining Act 1992* and the conditions of the licence.

In accordance with the *Mining Act 1992,* applications for the Mining Leases required for the mine, the processing and associated infrastructure have been made. The MLAs cover the entire Project area and can be converted to Mining Leases upon final application to the government.

The Company also owns a portion of the land in the Project area. The Syerston and Kingsdale properties represent two thirds of the total Project area. The Company is currently negotiating a Land Purchase Agreement over the Slapdown property. The Company also owns the Kelvin Grove property to the northeast of the Project area (over a portion of Platina Resources' Owendale Project), as well as the main portion of land over a limestone deposit, Westella to the east of the Project. The Company is currently in negotiating a Land Option Agreement over The Troffs property. All properties are currently tenanted (Figure 1-4).

The grant of a Mining Lease over freehold land requires, for practical purposes, the consent of the landholder. Scandium21 has purchased most of the land over the Project area to ensure that obtaining such consent will not delay the Mining Leases being granted. Land ownership provides a significant advantage for the approval of these Mining Leases.

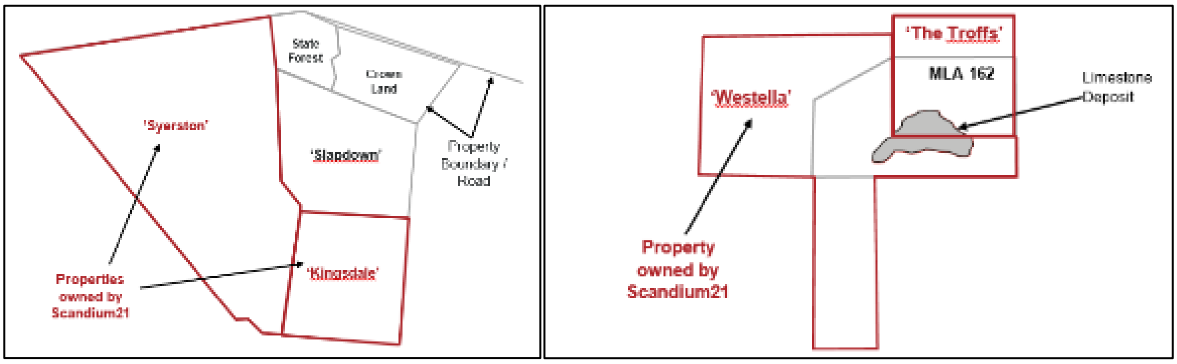



**Figure 1-4: Scandium21's landholding – main project area (left) and limestone quarry (right)**

## 1.3 Geology and Mineral Resource

### 1.3.1 Summary

The deposit lies over a mafic to ultramafic intrusive complex known as the Tout Complex. The lateritic weathering profile in which nickel, cobalt, scandium and platinum accumulated developed preferentially over the dunite core of the intrusion, which is about 4 km by 2 km in area.

Over the dunite, the laterite profile can be up to 35 - 40 m thick, but thins markedly over surrounding pyroxenites so that it has a basin-like form.

Several Tertiary drainage channels cut across the deposit. These reach depths of 20 - 25 m and are now filled with barren alluvium. The laterite profile is best developed on old Tertiary hilltops between these palaeo-drainages that divide the deposit into two main parts.

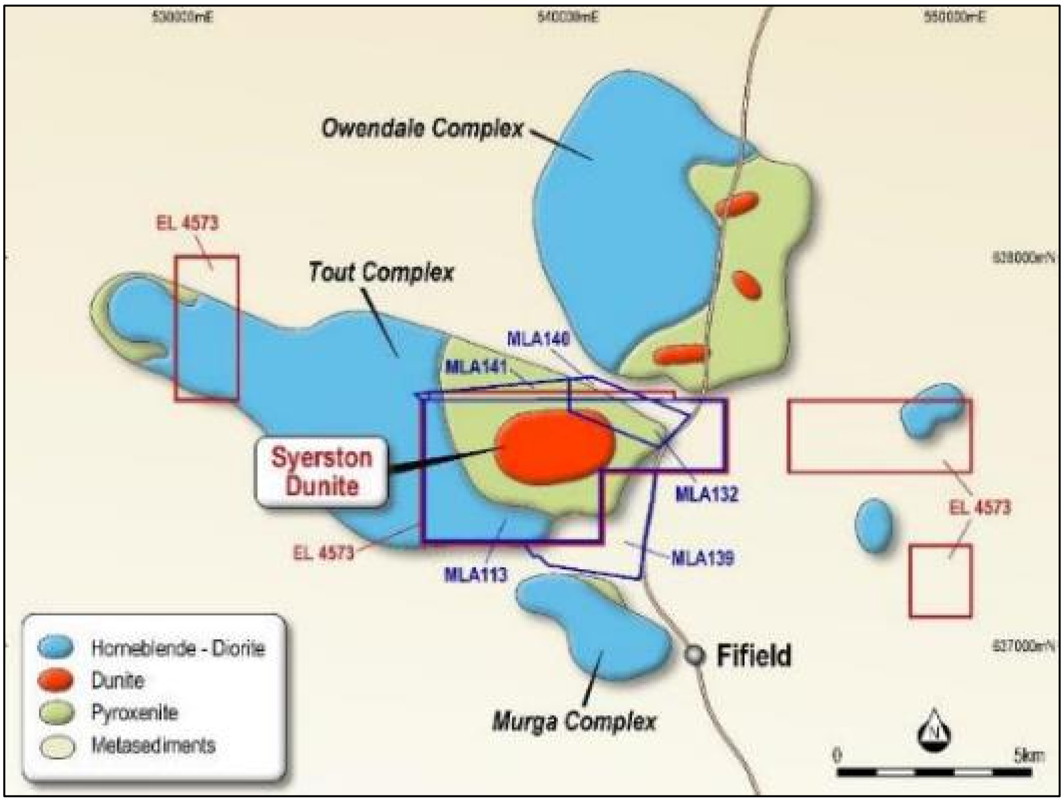



**Figure 1-5: Geology of the Syerston Project area**

### 1.3.2 2016 Resource model

The 2016 Mineral Resource described below was used as the basis for the Mineral Reserve. Historically, 1,200 drill holes have been drilled in several drill campaigns. Significant drilling was carried out on the dunite core, with minimal drilling along the periphery of the historical nickel/ cobalt deposit. A 2014 - 2015 drilling campaign focused on identified high grade scandium zones to the northwest of the deposit; however, these have not been included in this resource, as nickel and cobalt mineralisation present in these areas is very limited.

A 3D geological model was developed to cover the full extent of the interpreted high grade pods. Resources estimation was based on all data that was considered reliable. The majority of the nickel cobalt resource is in the Measured and Indicated Resource categories. The nickel and cobalt metal distribution in the main geological unit is shown in Figure 1-6.

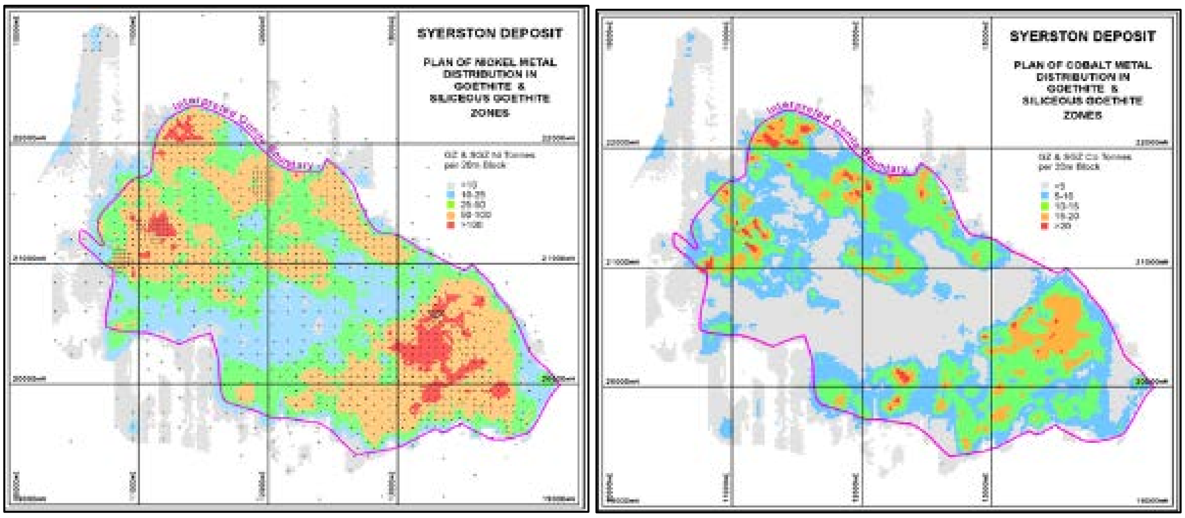



**Figure 1-6: Projected plan view of nickel and cobalt metal distribution**

Based on this modelling, a Mineral Resource estimate for nickel and cobalt was produced (Table 1-2).

**Table 1-2: 2016 Syerston Mineral Resource estimate at 0.6% NiEq cut-off**

| Cut-off NiEq % | Classification | Inventory (Mt) | NiEq* (%) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|---|
| 0.6 | Measured | 52 | 1.05 | 0.73 | 380 | 0.109 | 57 |
| 0.6 | Indicated | 49 | 0.87 | 0.58 | 280 | 0.101 | 49 |
| *0.6* | *Meas + Ind* | *101* | *0.97* | *0.65* | *660* | *0.105* | *106* |
| 0.6 | Inferred | 8 | 0.83 | 0.54 | 50 | 0.098 | 8 |

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes and contained metal are rounded to the nearest thousand.
3) Totals may appear different from the sum of their components due to rounding.
4) A cut-off grade of 0.6% NiEq, NiEq% = Ni% + (Co% x 2.95)
5) USD4/lb, cobalt at USD12/lb and a nickel recovery of 90% and cobalt recovery of 88.9%, USD:AUD of 0.75.
6) The Mineral Resource estimation was verified by Danny Kentwell, FAusIMM, who is a full-time employee of SRK Consulting. Danny Kentwell, FAusIMM, a full-time employee of SRK Consulting is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

### 1.3.3 2017 Resource model

A new database was developed for the 2017 standalone resource model, sourcing data from original historical material as far as possible. A revised geological interpretation was undertaken utilising the major element (Fe, Si and Al) and minor element (Ni and Co) chemistry to identify the chemical characteristics of the various zones within the laterite profile.

A 3D geological model was developed to cover the full extent of the cobalt-nickel laterite zones both within and outside the Dunite Complex footprint. In addition, high grade cobalt and scandium domains were separately generated. The high grade cobalt distribution in the main Goethite geological unit is shown in Figure 1-7. The high grade scandium distribution is shown in Figure 1-8.

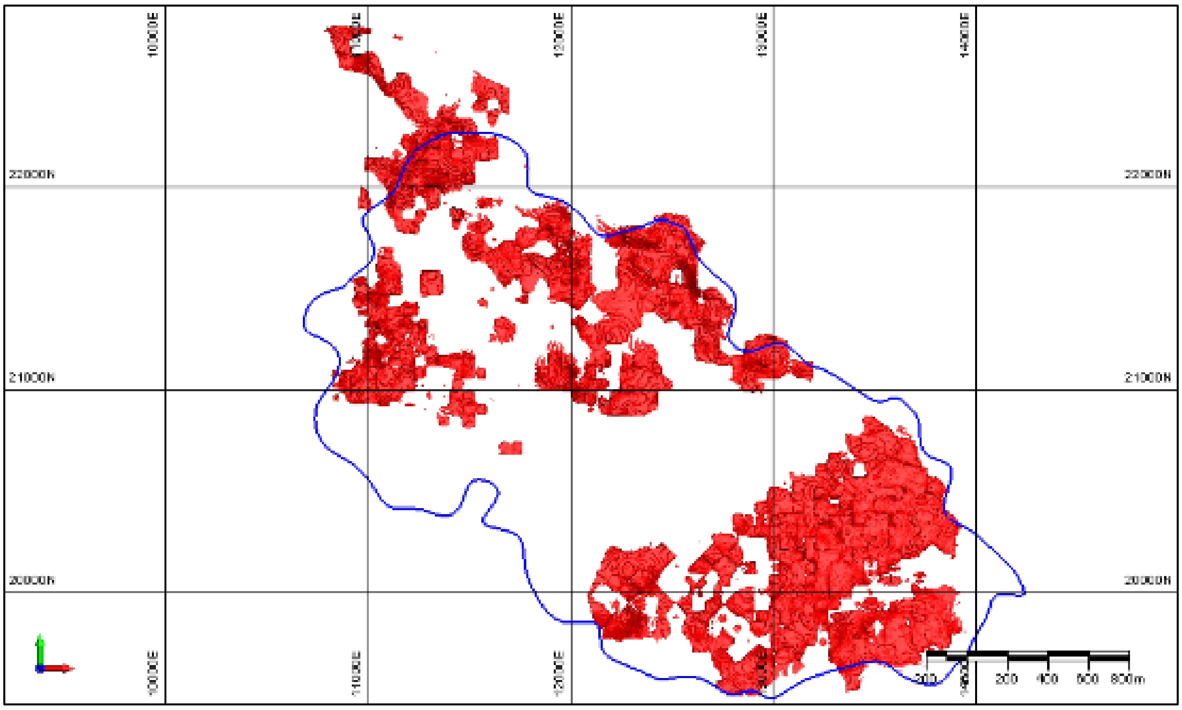



**Figure 1-7: High grade cobalt locations**

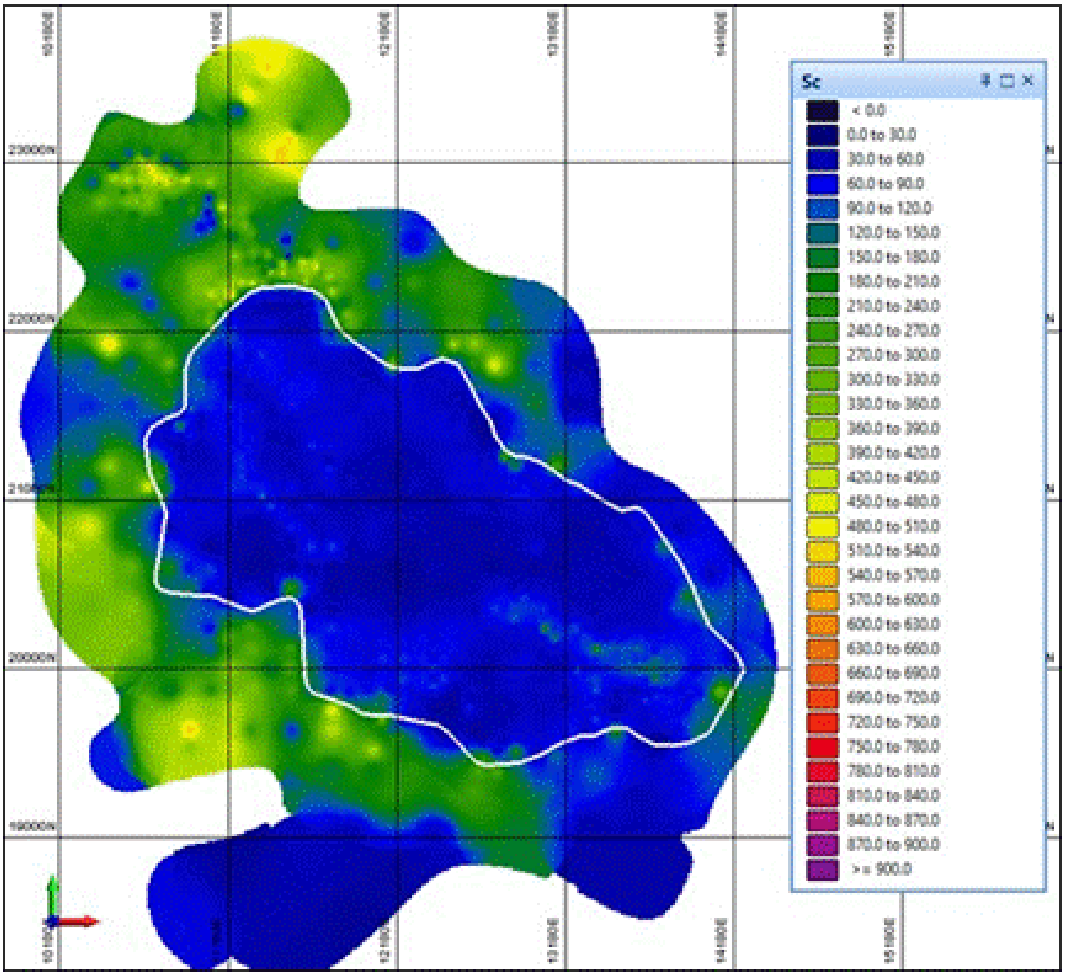



**Figure 1-8: Scandium grade distribution**

Resource estimation was carried out on domain basis, using Ordinary Kriging and a resource classification was developed based on data spacing and geological and statistical confidence. The majority of the nickel cobalt resource is in the Measured and Indicated Resource categories.

**Table 1-3: 2017 Syerston cobalt/ nickel Mineral Resource estimate at 0.06% Co cut-off**

| Cut-off (% Co) | Classification | Inventory (Mt) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|
| 0.06 | Measured | 40 | 0.75 | 299 | 0.15 | 59 |
| 0.06 | Indicated | 47 | 0.55 | 259 | 0.12 | 58 |
| 0.06 | Meas + Ind | 87 | 0.64 | 558 | 0.13 | 116 |
| 0.06 | Inferred | 14 | 0.24 | 34 | 0.11 | 16 |

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes and contained metal are rounded to the nearest thousand.
3) Totals may appear different from the sum of their components due to rounding.
4) A cut-off grade of 0.06% Co
5) The Co-Ni Resource partially includes material contained within the reported Scandium and Platinum Resources
6) The Mineral Resource estimation was prepared by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar & Associates. Lynn Widenbar, MAIG, a full-time employee of Widenbar & Associates is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

## 1.4 Mining and Mineral Reserve

Mining is planned to be undertaken by conventional open pit methods, using conventional excavators and trucks. Scandium21 will engage the services of a mining contractor to operate the open pits. The contractor will also be responsible for the mining related construction activities, including the run of mine (ROM) pad, limestone pad and haul road construction, and maintenance during operations. The actual equipment to be used will be agreed with the selected mining contractor, to allow ore selectivity within the pit and requirement for high manoeuvrability between the pits to be maintained.

Mining activities will begin with the development of roads to allow access to all areas of the deposit and material haulage in all weather conditions. This will entail the provision of a hard and suitably draining road base. Establishment of all-weather access roads will be followed by carefully managed clearing of the mine sites themselves. For the purpose of this study, it is assumed that the removal of vegetation will be carried out mechanically, followed by transport to a designated area.

### 1.4.1 Pit model and mine schedule

In the 2005 Study, pit optimisations were undertaken on the 2005 resource model with relevant dilution, cost, revenue and geotechnical inputs taken into consideration. The optimisation pit shells were used for detailed pit design, taking ramps and geotechnical considerations into account.

Pit designs were not prepared as a basis for this Report. This Report uses the 2005 open pit designs – it is considered that as there were limited changes to the resource, the updated pit optimisation is likely to be similar to the results of the 2005 Study. Waste dumps and stockpiles were designed to accommodate the volumes in the mine schedule.

SRK recommends that updated pit optimisations and pit designs be undertaken in the further study work to confirm the pit staging and to optimise the Mineral Reserve, ore and waste schedules.

SRK reviewed the work undertaken to support the 2016 PFS. SRK has verified and validated the outputs from the mining studies to satisfy itself of the integrity of the work.

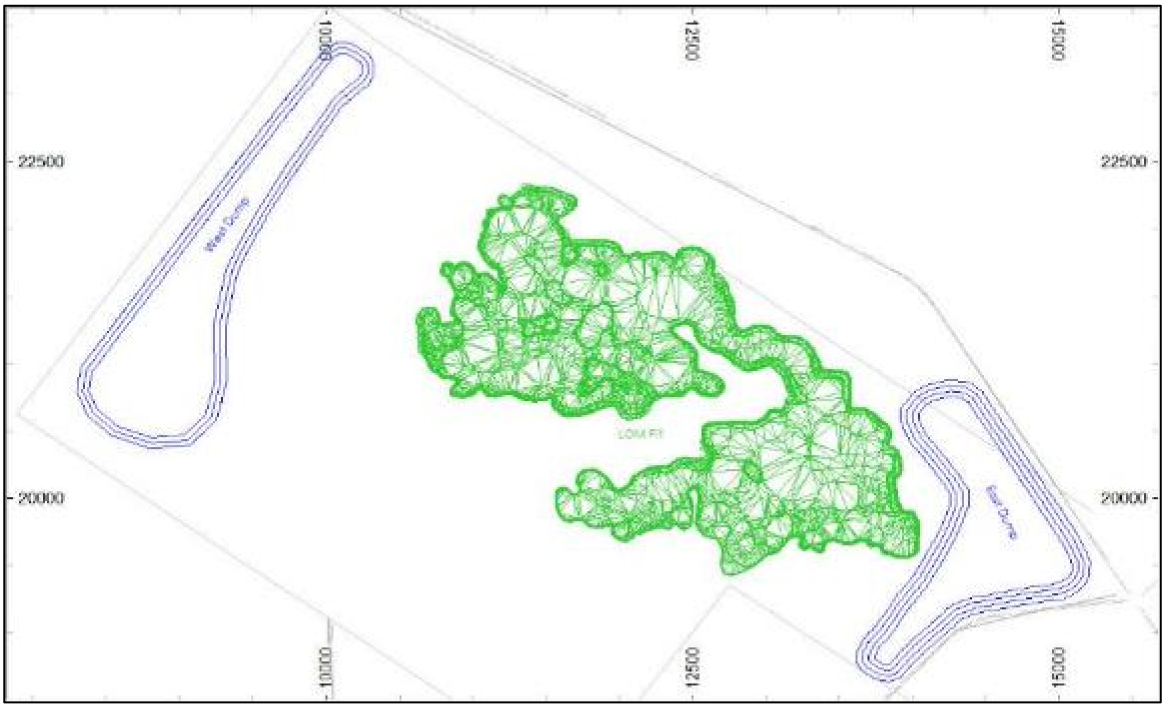



**Figure 1-9: Pit model for LOM operation**

The Mineral Reserves are based on a calculated block value and cut-off grade. Only Measured and Indicated Mineral Resources have been converted to Proved and Probable Mineral Reserves. The Mineral Reserves have been estimated using the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012, JORC Code (2012). The Mineral Reserves are shown in Table 1-4.

Peter Fairfield of SRK has completed a review of the relevant supporting information and examined relevant working files for validation and is satisfied that the validation checks and conclusions are correct as described. As the Qualified Person, Peter Fairfield takes responsibility for the reporting of the Mineral Reserves in this Report.

**Table 1-4: Mineral Reserve (as autoclave feed tonnes)**

| Category | Inventory (Mt) | NiEq (%) | Cont. Metal (NiEq kt) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|---|
| Proved | 59.48 | 0.96 | 571 | 0.71 | 422 | 0.1 | 59 |
| Probable | 44.23 | 0.83 | 367 | 0.58 | 257 | 0.1 | 44 |
| **Total** | **103.71** | **0.91** | 944 | 0.65 | 674 | 0.1 | 104 |

Notes:

1) Tonnes are rounded to the nearest thousand.
2) Totals may appear different from the sum of their components due to rounding.
3) A cut-off grade based on NSR was used of approximately 0,25% NiEq, NiEq% = Ni% + (Co% * 2.95).
4) USD 7.50/lb nickel and USD12/lb cobalt and nickel recovery of 90% and cobalt recovery of 88.9%, USD:AUD of 0.75.
5) For economic modelling, a cobalt price of USD14/lb was used and USD 7.20/lb for nickel.
6) The Mineral Reserve is a subset of the Measured and Indicated only schedule of a Life of Mine Plan that includes mining of Measured, Indicated and Inferred Resources.
7) The Mineral Reserve estimate was independently verified by Peter Fairfield, FAusIMM, CP (Mining), who is a full-time employee of SRK Consulting and a Qualified Person under NI 43-101.

Mining costs were provided from contractor submissions in 2016. The block value in the mining model was calculated by incorporating the estimated processing cost (fixed and variable), metal recoveries, metal prices and the average acid consumption cost for different rock types.

The net smelter return (NSR) was calculated as the revenue less operating costs (excluding mining). The metal prices used for the block value were USD7.50/lb Ni and USD12.00/lb Co. Scandium has not been included.

The Mineral Reserve is based on the marginal cut-off grade and a block value of AUD0.00/t, which equates to an NSR AUD50.00/t or 0.25% NiEq. For the purpose of maximising NPV in scheduling, each block was assigned a Low Grade (low grade), Medium Grade (medium grade) or High Grade (high grade) subcategory value.

The Project NPV was maximised by scheduling the high grade ore as plant feed for the first 10 years of the Project. Following this, stockpiled low grade and medium grade ore will also feed the plant until the completion of mining. Figure 1-10 shows ore will be fed to the autoclave at a rate of 2.5 Mtpa. Mining is estimated to be completed in Year 25, with the feeding of stockpiled material being completed in Year 44.

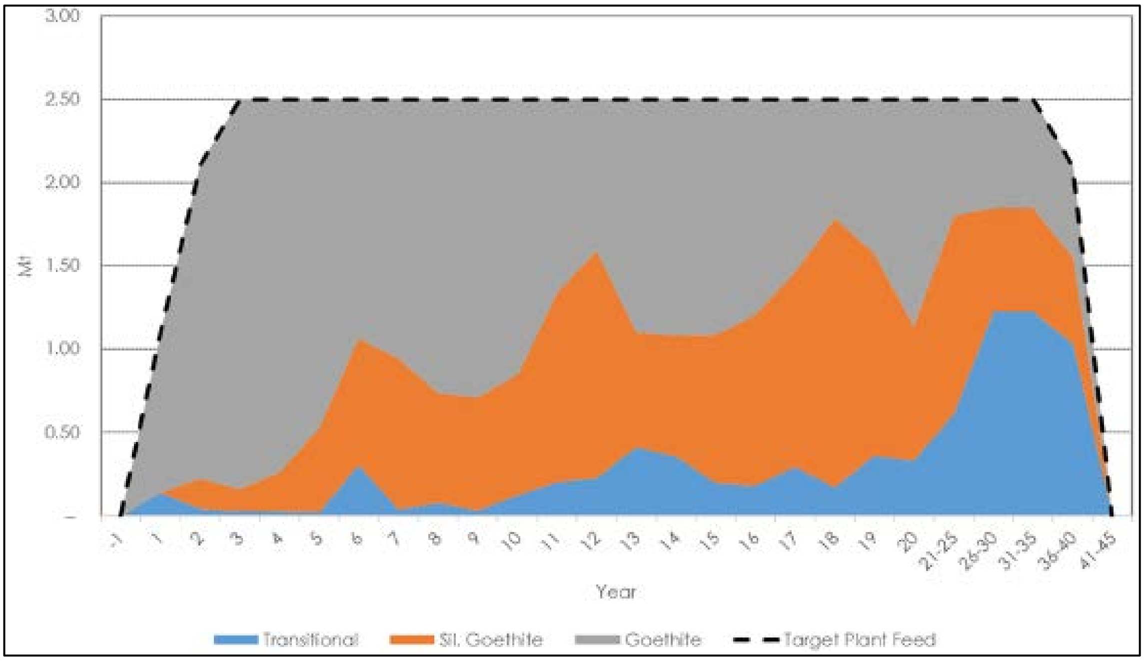



**Figure 1-10: Annualised LOM plant feed**

### Limestone supply from Westella quarry

Limestone feed is required for acid neutralisation at the plant. The Westella limestone deposit is located approximately 22 km to the southeast of the proposed Syerston Mine and Processing Facility. The design work undertaken in the 2005 Study for mining limestone from Westella was used for the 2016 PFS and the operating costs were updated in the 2016 PFS.

High grade limestone will be crushed and stockpiled at the quarry ready for transport by a mining contractor. Low grade limestone and waste material will be stockpiled separately, so that low grade limestone can be reclaimed at a later stage, if required.

It was identified that there would be sufficient high grade limestone feed in the current model for approximately 17 years. Scandium21 indicated that the resource continues to the north, but further drilling is required in order to prove this. For the purpose of the mining study, it was assumed that the limestone resource continues to the north, and all mining constraints remained constant, i.e. specific gravity (SG), strip ratio, etc.

## 1.5 Technology overview

### 1.5.1 High pressure acid leach

The Project has selected high pressure acid leaching (HPAL) and continuous resin-in-pulp (cRIP), along with downstream impurity removal, solvent extraction and crystallisation, as the processing route for the separate high purity nickel sulphate and cobalt sulphate products.

There is a high degree of confidence in the robustness of the HPAL process to extract nickel and cobalt from lateritic ores. The HPAL processing technology is generally considered to be in its fourth generation and is much improved by the experience gained from its use in operating plants. While challenges in commissioning and ramping up HPAL operations remain, these challenges are well understood. Commercial HPAL processing of laterites commenced at Moa Bay in Cuba in the late 1950s. Since that time, several large HPAL plants have been constructed and successfully operated on laterites for nickel and cobalt extraction. These include plants at the Bulong, Cawse, Murrin Murrin, Ravensthorpe, Coral Bay, Ambatovy, Taganito and Goro operations.

The metallurgical testwork completed in the two previous feasibility studies on the Project typically followed nickel, cobalt and scandium. All historical testwork confirms that scandium extraction using the HPAL process ranges from 80% - 90%, and is sometimes higher.

### 1.5.2 Resin-in-pulp

Clean TeQ uses a proprietary ion exchange technology (Clean-iX®) for the extraction and purification of metals. The base technology for the Clean-iX® process was developed by the All Russian Research Institute of Chemical Technology (ARRICT) over a period of 40 years.

Since 1951, ARRICT has been involved in the development of over 30 mining operations using the technology, mainly for uranium and gold extraction, from leached slurries and solutions. In 2000, Clean TeQ obtained the exclusive licence for all technical information relating to ion exchange resin, ionic membranes, organic solvent extractants, including manufacturing know-how and plant design, for all countries outside the former Soviet Union. Since obtaining the licence, Clean TeQ has further developed the technology for base metals, uranium and gold, with particular improvements in relation to laterite ore processing, scandium and uranium. Clean TeQ has been granted 10 additional patents on various aspects of the technology, including one for extraction and purification of scandium.

The application of resin-in-pulp (RIP) for scandium recovery is based on commercially applied (for other metals) and developed equipment and technologies and has the potential to offer significant benefits for nickel, cobalt and scandium recovery at the Project. Conventional flowsheets in the industry currently use counter current decantation (CCD) followed by precipitation of either a mixed sulphide or hydroxide intermediate, re-leaching and solvent extraction (SX) to recover the metals from leached slurries. This process has several disadvantages, including higher capital and operating costs, and lower metal recoveries. The use of RIP technology addresses many of these issues. The RIP method uses solid ion exchange resin beads which are contacted directly with the leached slurry, resulting in extraction of more than 98% of the contained metal in the solution. Ion exchange resins are ideal for recovery and concentration of lower concentration metals, which is the case with lower grade laterite resources. A benefit is that the plant size and chemical reagent costs are reduced in comparison with SX.

Clean TeQ's development of the nickel, cobalt and scandium RIP process since 2001 includes three large-scale piloting operations on laterite ore. Furthermore, Clean TeQ has developed a process for scandium recovery from titanium dioxide process streams and has a fully automated pilot plant operating on such process streams.

### 1.5.3 Metallurgical testwork summary

Extensive metallurgical piloting on the extraction and recovery of nickel and cobalt was completed by the two previous owners and included variability testing of more than 100 composites of different ore lithologies. This work has provided a solid basis on which to establish the design criteria for the nickel/ cobalt/ scandium Project. During each of these testwork programs, nickel and cobalt were the primary targets, but scandium was also analysed. This work provides a relatively high degree of confidence on metal extraction using the HPAL process and subsequent unit process design criteria. In addition to this earlier work, basic sighter tests were carried out to confirm the results of previous metallurgical testwork.

Clean TeQ's more recent metallurgical testwork and processing development focus was applying the RIP technology, a change to the original processing flowsheet, to the HPAL discharge slurries.

The Syerston ores have been shown to have relatively low acid consumptions, typically 240 - 290 kg/t. This is a key advantage for the Project as acid consumption makes up a significant proportion of the total operating costs. HPAL testing has also demonstrated high leach extractions – over 97% and

95.5% for nickel and cobalt respectively – and higher overall metal recoveries.

## 1.6 Processing plant

A complex hydrometallurgical processing flowsheet using conventional HPAL to leach nickel and cobalt from the Syerston ores will be used at the proposed Syerston Processing Plant. The leached nickel and cobalt is then recovered through nickel and cobalt RIP and SX before the final nickel sulphate and cobalt sulphate products are crystallised, dried, packaged and transported to market. Optionally, a third scandium oxide product can be produced from the raffinate liquor streams, again using scandium RIP, scandium refining and final calcination.

Figure 1-11 shows an overview of the processing options considered for this study. Although the optional scandium oxide production option is shown, this option is not currently incorporated in the detailed flowsheet that forms the basis of the capital and operating costs.

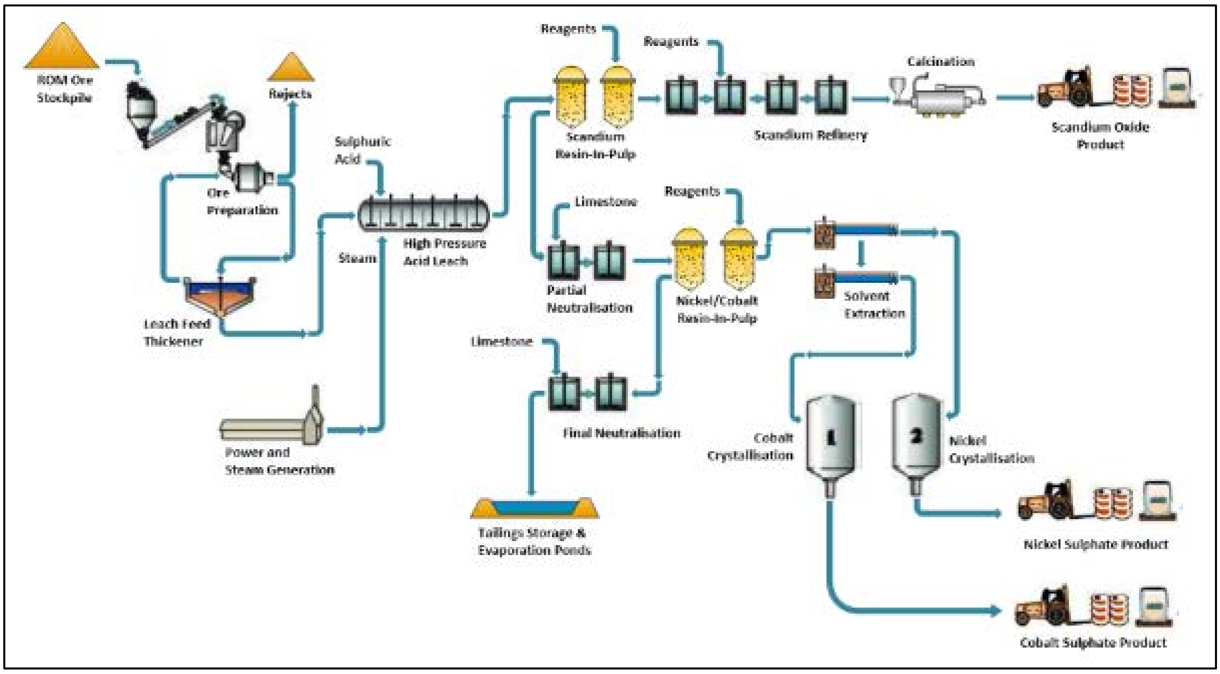



**Figure 1-11: Process flowsheet**

Detailed descriptions of each process are provided in Section 17 (Recovery Methods). The process can be broadly defined as:

- Ore Preparation and Milling
- HPAL
- Scandium RIP (optional, but not included in cost estimate)
- Scandium purification (optional, but not included in cost estimate)
- Nickel/ cobalt RIP
- Nickel/ cobalt sulphate purification and recovery
- Tailings neutralisation and storage
- Process reagents and utilities (sulphuric acid, steam, water, limestone, other).

The final slurry/ solution after metal recovery is neutralised with limestone and sent to a tailings storage facility (TSF) and an evaporation pond. The process plant will produce high purity hydrated nickel and cobalt sulphate products, as well as a 99.9% $Sc_2O_3$ product (optional).

### 1.6.1 Hydrated nickel and cobalt sulphates

The process plant has been designed to produce high purity hydrated nickel sulphate ($NiSO_4.6H_2O$) and hydrated cobalt sulphate ($CoSO_4.7H_2O$) products. The product from the elution circuit of the RIP plant is a high concentration, high purity combined nickel and cobalt sulphate solution. Therefore, the process is ideally suited to the lithium ion (Li-ion) battery sector, which requires sulphates for precursor production, and potentially eliminates process steps that exist in the current cathode supply chain.

Based on historical data taken from the 2005 Nickel/ Cobalt Feasibility Study Update, the 2016 Scandium Feasibility Study and Clean TeQ's internal database of nickel and cobalt recovery using the RIP process, the process design criteria (PDC) were developed for the 2016 PFS. Table 1-5 provides a summary of the key process parameters for each option over the initial 20-year mine life.

**Table 1-5: Process design criteria summary**

| | **tpa** | **2,500,000** |
|---|---|---|
| Operating hours per year | hours p.a. | 7,690 |
| Availability Leach Plant | % | 89.6 |
| Mine Life | years | >20 |
| Average Product Production (Years 3 - 20) | | |
| Nickel Sulphate ($NiSO_4.6H_2O$) | tpa | 85,136 |
| Cobalt Sulphate ($CoSO_4.7H_2O$) | tpa | 15,490 |
| Nickel Equivalent Production | tpa | 18,730 |
| Cobalt Equivalent Production | tpa | 3,222 |
| *Scandium Oxide ($Sc_2O_3$) - Optional* | *tpa* | *~ 50* |
| Nickel Grade (Average) | % | 0.80 |
| Cobalt Grade (Average) | % | 0.14 |
| Nickel Grade (Range) | % | 0.7 - 1.0 |
| Cobalt Grade (Range) | % | 0.1 - 0.2 |
| Scandium Head Grade (Range) | ppm | 40 - 60 |
| Autoclave Operating Temperature | ˚C | 250 |
| Autoclave Residence Time | minutes | 70 |
| Sulphuric Acid to Leach | kg/t | 240 - 290 |
| Estimated Leach Extractions (Goethite) | | |
| Nickel | % | 97 |
| Cobalt | % | 95.5 |
| Scandium | % | 86 |
| RIP Metal Recovery (Ni/ Co & Sc) | % | 98 |
| Estimated Overall Recoveries (Goethite) | | |
| Nickel | % | 93.5 |
| Cobalt | % | 92.7 |
| Scandium | % | 83.4 |

The mine production schedule is based on the 2005 Feasibility Study Update, in which production of cobalt was capped at 5,000 tpa, as the main focus was nickel production. However, there is potential to alter the mine plan such that higher grade cobalt is mined, particularly in the early years of the operation.

### 1.6.2 Infrastructure

One of the Project's competitive advantages is its proximity to existing infrastructure. The Project and associated infrastructure are located within the shires of Lachlan and Parkes, while the borefield providing most of the raw water for the Project is located in the Forbes Shire.

The proposed Project infrastructure facilities are representative of those required to support a modern 2.5 Mtpa complex hydrometallurgical plant and open pit mining operation. The Project is located relatively close to the Moomba–Sydney (natural gas) pipeline, a rail line is located within 20 km of the Project, giving it access to the ports of Sydney and Newcastle. Major bituminised arterial roads provide good access to the site. The major city and town centres have excellent infrastructure, including transport, airport and rail facilities, all of which are available to service the requirements of the Project. The workforce would be accommodated in the local communities.

The Project infrastructure comprises the following:

- Access road, internal roads and haul road
- Rail siding
- Integrated power station, high pressure steam boiler and acid plant
- Site buildings - office and administration complex, workshops, stores, ablutions and change house, fences and security
- IT and communications systems
- Sewage plant
- Store and laydown facilities
- Ore stockpiles and waste stockpile area
- ROM stockpiles
- Processing plant and associated facilities
- Raw water storage to manage rainfall runoff
- Process plant and mining workshops
- Tailings storage facilities
- Evaporation ponds
- Borefield
- Reverse osmosis plant
- Analytical and metallurgical laboratory
- Mobile equipment
- Diesel fuel storage.

The bulk of the raw water demand will be sourced from the Project's borefield 65 km south of the mine which is licenced for an abstraction rate of 3.2 GL/annum. This amount covers water requirements in terms of potable water, fire water, high pressure hose-down water, mine utility water and plant water for use in the process, as well as feed to the water treatment plant producing high purity water for steam production. During the 2005 FS, the volume of raw water from the borefield was not sufficient to meet the Project's entire raw demand and it was intended that this was to be supplemented with water taken directly from the Lachlan River. At the time, approvals were not obtained for this supplementary water source. A detailed Project water balance has not been updated as part of the 2016 PFS; however, this work is being undertaken as part of the ongoing FS. While the current project is reducing water demand, it is likely that supplementary water will be required. Alternative sources of supply may need to be considered if the appropriate approvals are not obtained in the future. Clean

TeQ plans to finalise the water the water balance as part of the FS before engaging a water broker to secure any additional water requirements.

Natural gas is transported to site by means of a new natural gas pipeline. Gas is used as an energy source for heating water to convert to high pressure steam used in the leaching process, as well as for electricity, via a gas-fired generator. The pipeline branches off from the existing Moomba–Sydney natural gas pipeline, passes near Condobolin and runs north to deliver natural gas to a metering station at the southwest corner of the Syerston process plant. The Project is not sufficiently advanced to enter into formal negotiations for the supply or transmission of natural gas. Due to the quantity required, it is more likely that supply will be arranged through a retailer/ gas aggregator rather than the primary producer. Gas suppliers, such as AGL, Alinta and Origin, have confirmed that there is sufficient capacity in the pipeline to supply the Project, which is in the order of 2.4 TJ/day of firm (not interruptible) supply. The APA Group (the owner and operator of the pipeline) has been re-engaged regarding the new pipeline and has confirmed its ability to build and operate, but the APA Group has not yet provided pricing, which will be based on the natural gas demand established in the FS engineering work. The pipeline operator and gas providers advised that insufficient gas availability will not be a risk to the Syerston Project, but pricing is less certain and there is a risk of escalating prices.

Sulphuric acid is produced on site with a dedicated acid plant capable of making 2,700 tpd of sulphuric acid. This incorporates a sulphur receival system incorporating sulphur handling, stockpile and reclaim areas designed to receive sulphur delivered by end-tipping road trains at regular intervals. Sulphur would be railed to a local siding located between the towns of Trundle and Fifield and loaded into end-tipping road trains. The acid plant will also produce high pressure steam at 6,000 kPa and 450°C. This steam is exported to the power plant where it is desuperheated for distribution to the process plant users. Any surplus steam is used in the power plant for power generation, and provides the potential to offset a portion of the site's net energy requirements.

Limestone will be mined in the Company's limestone quarry and trucked to site. Limestone is required for neutralising the process slurries and liquors following acid leaching. In order to meet this requirement, the original FS approvals to mine a maximum of 600,000 tpa of limestone from the Gillenbine limestone deposit, situated approximately 20 km southeast of the Project, were obtained. The approved design for the quarry presented in the environmental impact statement (EIS) includes the removal and stockpiling of waste rock and limestone extraction using conventional open pit drill and blast methods. Waste rock and low grade limestone would be deposited in an emplacement surrounding the open pit. All other chemicals will be trucked or railed to site in either B-Doubles, ISO containers, intermediate bulk containers (IBCs) or bulk bags and stored on site in a dedicated reagents area.

The infrastructure design and development has largely been undertaken during the earlier engineering feasibility studies for a 2.0 Mtpa (2000) and 2.5 Mtpa (2005) nickel and cobalt operation completed by the previous owners, Black Range and Ivanplats, respectively. Work was continued by Clean TeQ in 2016 and 2017, specifically associated with the application to modify the Project's 'Development Consent', which has now been given. This included Voluntary Planning Agreements (VPAs) with the local Shires outlining the contributions the Project would make in terms of road upgrades, road maintenance and contributions to community-based activities. It also updated other aspects of the Development Consent affecting local community stakeholders, including a review of several key aspects of the infrastructure – public roads, railway sidings, limestone quarry and the natural gas pipeline.

Because of the limited change in the infrastructure for the Project, the capital cost estimate associated with the infrastructure has been factored (by Clean TeQ), based on a SNC-Lavalin cost escalation assessment undertaken in August 2016. Additional allowances have been made for items excluded in previous studies, specifically to upgrade the local road network. A number of other costs have been excluded as these will be covered under a build-own-operate (BOO) or supply type arrangement. Firm pricing for these costs has not yet been obtained.

## 1.7 Environment, permitting, social and community

The Development Consent DA 374-11-00 has been modified on three occasions since it was issued in 2005. The most recent modification (in 2016) allows mining and processing operations to initially focus on scandium oxide production and gives approval for adjustments to the processing operations to allow for the production of approximately 80 tpa of scandium oxide and up to 40,000 tpa of nickel and cobalt metal equivalents, as either sulphide or sulphate precipitate products.

Details of the most recent proposed project modifications were set out in a document entitled, Syerston Project Scandium Oxide Modification Environmental Assessment (Document No. 00740462, May 2016). The modified development application was approved on 12 May 2017, subject to a range of conditions set out in the Notice of Modification. SRK considers that stakeholder comment on the recent development approval has generally been supportive of the Project and SRK considers it unlikely that third party appeals will arise.

The primary environmental consent required for project implementation has been granted. Based on a review of decisions for comparable recent developments in the general locality, SRK has no reason to expect that the secondary approvals required for the Project would be refused.

Scandium21 has been granted EL 4573; the licence is due to expire on 16 August 2018, but can be renewed, subject to compliance with the provisions of the *Mining Act 1992* and the licence conditions. The licence confers exclusive rights to prospect in the exploration area for Group 1 minerals, including rare earth minerals, nickel, cobalt and platinum.

Virtually all of the land required for project implementation is freehold land (much of it owned by Scandium21). As a result, there is minimal risk of any future exposure to Native Title claims.

The company has signalled its intention to acquire additional freehold land, including properties known as Slapdown and The Troffs in order to secure its interests over the proposed mine operations area and limestone quarry. At the time of reporting, Clean TeQ has an agreement in place (documented and binding, though not yet completed or settled) to purchase Slapdown and was negotiating an option agreement for The Troffs – this is also not yet completed.

Easement agreements are proposed in order to secure land required for the gas pipeline. Land option agreements and land purchase agreements are proposed in order to secure the land required for the rail siding and proposed bypass road around the town of Fifield. At the time of reporting, Clean TeQ had agreed terms to acquire the land that is the subject of the rail siding (documented and binding, though not yet completed or settled).

### 1.7.1 Water supply and borefield

Previous water investigations by Coffey in 2000 determined that insufficient water was available in the immediate Project area to meet the historical plant requirements. The closest viable source of sufficient water was the Lachlan River, approximately 65 km to the south of the Project area. Black Range and Ivanplats completed the EIS and Development Consent on the basis of this borefield being established.


**Figure 1-12: Borefield overview**

Clean TeQ Metals engaged Golder Associates Pty Ltd in March 2015 to determine the current status of water supply in the area. The Golder report concluded that there were no single groundwater or township water source that could meet the required demand. Therefore, the most practical water source reaming for the Project is the established borefield.

On 13 June 2006, the NSW Office of Water granted Water Bore Licences to Ivanplats Syerston (former company name of Scandium21) for the extraction of 100 L/s from the Project borefield.

The Eastern and Western borefields were established on the assumption of a duty/ duty/ standby arrangement. To date, one bore in each of the Eastern and Western borefields has been developed, with the Western borefield operating (for local use under a water purchase agreement).

SRK notes that the currently approved volume of water sourced from borefields is not sufficient to meet the Project's entire raw demand and the intention was to supplement with water taken directly from the Lachlan River. At the time of writing this Report, approvals had not been obtained for this supplementary water source. While the water demand current for the Project is currently reducing, it is still likely that supplementary water will be required.

As part of the Feasibility Study, Clean TeQ plans to finalise the water balance as well as assess options to secure rights to any additional water requirements.

### 1.7.2 Community and stakeholder consultation

A community consultation program will be undertaken for the Project, considering directly affected landowners; surrounding landowners; community groups, businesses and Aboriginal associations; Local Government; and Government Departments and Agencies.

The consultation program aims to keep the community informed about the development of the Project and to provide a means for stakeholders to comment on Project-related issues.

Currently, Clean TeQ has negotiated VPAs with each of the shires affected – Lachlan, Parkes and Forbes – to outline the contributions to community enhancement, road upgrades and ongoing road maintenance once the Project has commenced.

Once the Project progresses to the next stage of development, the community consultation program will be re-initiated.

## 1.8 Nickel and cobalt market

While there are established markets for nickel (stainless steel, alloys and plating) and cobalt (chemicals, super alloys, catalysts), a significant growth sector for both metals is in Li-ion batteries. Growth in the Li-ion battery market is driven by the increased consumption of portable electronic devices (mobile phones, laptops, tablets), the electric vehicle (EV) market and growing applications for utility energy storage. Cobalt and nickel are used in the majority of cathode chemistries available, with a growing shift towards nickel-cobalt-manganese (NCM) and nickel-cobalt-aluminium (NCA) over chemistries that do not use nickel or cobalt. Lithium-cobalt-oxide (LCO) chemistry remains the chemistry of choice for many applications in the portable consumer electronics market, given its high-energy density.

The production of cathode precursor material for Li-ion batteries typically requires the processing of nickel and cobalt in the form of hydrated sulphates, being $NiSO_4.7H_2O$ (~22% Ni) and $CoSO_4.6H_2O$ (~20% Co). In NCM chemistry, manganese compounds are also required.

### 1.8.1 Pricing

Both the nickel and cobalt markets have established trading platforms, either through the London Metals Exchange (LME) or Metal Bulletin. However, sulphate product pricing tends to be more opaque, due to the relatively small size of this sub-market. Sulphate pricing mechanisms are typically determined on a premium to either LME (nickel) or Metal Bulletin (cobalt), reflecting the additional cost to convert nickel and cobalt-bearing materials into high purity sulphates. These premia can often be high, but appear to vary substantially depending on the market and product quality.

The Li-ion battery industry has not been a significant consumer of nickel and, more importantly, cobalt, but this may change in the future. As production capacity is filled, raw material costs, particularly associated with nickel and cobalt, will become more important in the total battery production cost. Cathode costs represent the largest raw materials cost in the production of Li-ion cells. Supply of critical metals for the battery sector, particularly cobalt, will require new sources of production to keep up with growing demand from the Li-ion battery industry.

The 2016 consensus long-term metal prices from market analysts, such as CRU International Ltd (CRU) and CIBC, forecast nickel and cobalt prices to rise over coming years, finding equilibrium again at historic long-term average prices. These forecasts predict nickel LME prices to rise to USD7.50/lb and cobalt prices to increase to USD12.00/lb. SRK notes that the pit optimisation work was conducted using a nickel price of USD4.00/lb.

For the design purposes of this study, long-term pricing assumptions have been adopted. Additionally, no set premia for nickel and cobalt have been assumed for the production of sulphate products, as the quantum of these premia over the longer-term is unclear. However, it is important to note that the premia represent (potentially material) upside to pricing outcomes. Further work is being undertaken with industry consultants to evaluate the long-term quantum of potential premia for sulphate products over and above metal prices.

## 1.9 Financial summary

The capital and operating costs for the processing options are summarised in the following sections. The costs have also been provided in USD, using an exchange rate of 0.75USD:1.00AUD.

For economic analysis, metal prices are based on the long-term average consensus price forecasts from CIBC (as at August 25, 2017) for a range of analysts. The long-term consensus nickel price was USD9.00/lb. However, a discount to this price has been assumed for this study and a price of USD7.50/lb has been used to assess the economics of this Report. The long-term cobalt price was USD14.00/lb and has bene used to assess the economics of this Report. As sulphate products are being provided and typically trade at a premium, assuming long term LME and LMB prices for nickel and cobalt respectively is considered to be conservative.

### 1.9.1 Capital and operating cost estimate

A capital cost estimate for the design and construction of a 2.5 Mtpa HPAL processing plant incorporating RIP processing was developed in-house by Clean TeQ at a PFS level of accuracy. It was based on the earlier September 2005 FS capital cost estimate completed for the Project by SNC-Lavalin (Australia) and JGC Corporation (Japan). These costs were adjusted based on the review and further modifications made, largely the inclusion of a RIP circuit, nickel and cobalt sulphate refinery, road upgrade allowance and adjustments to other areas, based on the changes associated with the modified flowsheet.

An FS has now been initiated to develop the full plant costs from first principles, but as an interim measure, to further improve the confidence in the capital costs, Clean TeQ has further developed the RIP plant costs from first principles and has engaged Simulus Engineers (Simulus) to develop the refinery capital costs to better reflect the downstream processing circuits, i.e. post Ni/Co RIP, to a PFS level of accuracy. These costs have been built up using mechanical equipment costs, material take-offs for civil and structural steelwork, piping and valves, electrical load list from the mechanical equipment, with other engineering discipline costs such as instrumentation and control factored.

The HPAL section of the plant represents the largest cost centre. However, the use of RIP for nickel and cobalt recovery significantly reduces the size of downstream operations and the associated capital costs. The capital cost is estimated at AUD1,045M, based on the updated PFS cost estimates. Costs were based on an engineer-procure-construct-manage (EPCM) basis, incorporating a 15% contingency and are considered to be within the ±25% accuracy band expected of a PFS level of study accuracy (the previous PFS used a 10% contingency). The base date of the cost estimate is Q3 2017 and excludes the scandium recovery and purification circuits.

**Table 1-6: Capital cost estimate summary – 2.5 Mtpa**

| Plant Area | Cost | |
|---|---|---|
| **Currency** | **USDM** | **AUDM** |
| Mining | 14 | 18 |
| Site Preparation | 11 | 15 |
| Process Plant | 210 | 280 |
| Process Utilities | 85 | 113 |
| Services | 110 | 146 |
| Infrastructure | 45 | 60 |
| **Total Directs** | **474** | **632** |
| Indirects, including EPCM | 105 | 140 |
| Owners Costs, including spares and first fills | 103 | 137 |
| **Capital Cost, excluding contingency** | **682** | 909 |
| Contingency (15% of Directs/ Indirects) Less Duty | 102 | 136 |
| **Total Capital Cost Estimate** | **784** | **1,045** |

The capital cost estimate excludes a number of supporting infrastructure costs. These are expected to be provided on a BOO basis or direct supply cost, and are not incorporated into the capital cost estimate. The following exclusions are noted:

- Mining operation including fleet, buildings and workshops
- Limestone from owner's quarry based on a per tonne supply cost
- Supply of all reagents, including sulphur port and rail facilities

- Liquid nitrogen plant
- Mobile plant and equipment required for operations
- Site mobile vehicles plant
- Natural gas pipeline.

Re-engagement and initial discussions have been held with a number of the key third providers; however, the technical details and commercial terms have not been finalised, largely due to the level of study currently being undertaken. In most cases, this leverages the engineering work currently undertaken during the 2005 FS. Firmer pricing will be developed during the next phase of study as part of more formalised negotiations.

Clean TeQ developed operating cost estimates to support the Syerston Nickel Cobalt PFS, based largely on the work undertaken during the 2005 FS. The estimates were originally built up from first principles. Input values were updated to reflect new contract mining, reagent, utility prices and labour costs. The stated accuracy of the estimate is reported to be within ±20% as at Q4 2016. The claimed accuracy range is consistent with PFS guidelines.

The operating cost establishes the Project as potentially one of the lowest cost nickel producers on the market, especially after considering the by-product cobalt credits. As Syerston has a relatively high proportion of cobalt compared to other laterites, the total cost of nickel production is relatively low. No scandium by-product credits have been assumed in the base case.

The annual operating costs for the Project over the first 20 years of operation are summarised in Table 1-7. The costs are based on Years 3 - 20 as tonnage is still ramping up in Years 1 and 2. The operating cost includes mining, processing, production and logistics to offtake customers. An exchange rate of USD0.75:1.00AUD was assumed.

**Table 1-7: Operating cost summary (Years 3 - 20) – product sea freight**

| Cost centre | Total cost (AUDM) | Total cost (%) | Unit cost (AUD/t ore) | Unit cost (USD/lb Ni) | Unit cost (USD/lb Ni) after Co credits |
|---|---|---|---|---|---|
| Mining | 39.6 | 20.7 | 15.86 | 0.72 | |
| Processing | 142.3 | 74.2 | 56.90 | 2.58 | |
| Utilities | 0.1 | 0.1 | 0.03 | 0.00 | |
| Services & Infrastructure | 0.8 | 0.5 | 0.34 | 0.02 | |
| Finance & Admin | 8.9 | 4.6 | 3.56 | 0.16 | |
| Total | **191.7** | **100** | **76.69** | **3.48** | **1.42** |

### 1.9.2 Project valuation

Project valuation has been completed using the discounted cash flow (DCF) methodology. A cash flow model was constructed for the three options, based on inputs from the technical model/ mass balance and the engineering estimate completed by SNC- Lavalin in 2005 and Clean TeQ for RIP.

The Project considers a 21-year project life, with Years -1, -2 and -3 as the construction period, and metal production occurring between Years 1 and 20. The model development is based on an estimate of the real cost of operations and therefore excludes any allowance for inflation.

The key inputs and assumptions are presented in Table 1-8.

**Table 1-8: Syerston valuation model key inputs**

| Parameter | Unit | Value |
|---|---|---|
| Autoclave Throughput | tpa | 2,500,000 |
| Average Production (20-year average) | | |
| Nickel Sulphate ($NiSO_4.6H_2O$) | tpa | 82,469 |
| Cobalt Sulphate ($CoSO_4.7H_2O$) | tpa | 15,168 |
| Recovery (20-year average) | | |
| Ni | % | 94.2% |
| Co | % | 93.0% |
| Life of Mine | years | 39 |
| Nickel Long-term Price | USD/lb | 7.50 |
| Cobalt Long-term Price | USD/lb | 14.00 |
| Exchange Rate (AUD: USD) – Life of Mine | 1 : n | 0.75 |
| Discount Rate | % | 8% |
| Tax Rate | % | 27.5% |
| Royalties | | |
| NSW Government | % | 4% |
| Ivanhoe Mines (after Govt royalty) | % of Revenue | 2.5% |
| Depreciation | % | 20% declining balance |
| Sustaining Capital (all years) | % of Directs | 1.25% |
| Plant Financing Strategy | - | 100% Equity |

### 1.9.3 Discounted cash flow analysis

Table 1-9 summarises the discounted cash flow valuation for the Project for the life of mine based on the assumptions in Table 1-8.

**Table 1-9: Discounted cash flow valuation**

| Parameter | Unit | Value |
|---|---|---|
| NPV (post-tax) | AUDM | 996.7 |
| | USDM | 747.5 |
| Internal rate of return (IRR) (post-tax) | % | 21.0 |

A sensitivity analysis was run for – feed grade, nickel price and payability have the most significant impact on the economics of the Project.

## 1.10 Risks and opportunities

The risks and opportunities identified at the Project are listed in Table 1-10.

**Table 1-10: Project risks and opportunities**

| Risk/ Opportunity | Summary |
|---|---|
| Technology | • The HPAL process is now in its fourth generation, with plants successfully operated since the 1960s. Many of the learnings of previous operations were incorporated in the 2005 FS. Since that time, a number of new operations have commenced. Suitably qualified engineers will be used for any revised design work for the HPAL system. Similarly, RIP technology has successfully been implemented since the 1950s on over 30 full-scale operations to recover a range of metals. In the laterite context, Clean TeQ has spent over 13 years developing and piloting the technology on laterite ores for optimal recovery of nickel, cobalt and scandium.<br>• To minimise technical risks, a well-resourced owner's management team with high levels of expertise gained from direct exposure to developed lateritic nickel projects, will be established. The resource and mining risk relating to the Project is deemed to be low because of the continuity of the deposit and the use of well-established resource estimation techniques. Resource and mining reserve estimates established for the first generation HPAL projects have proven to be reliable.<br>• Land access/ acquisition agreements will need to be finalised to cover parts of the Project area, limestone quarry, rail siding and natural gas pipeline. Additionally, the current water allocation will need to be increased or additional licences purchased to secure finalise the Project's water requirements. The water requirements are yet to be finalised and the final water access agreements are yet to be confirmed. |
| Political | • The key permitting constraint identified by SRK for the Project relates to consents required to source and use water (whether surface water or groundwater) to satisfy operational water requirements. The primary Development Consent development consent required for project implementation was granted in May 2017, subject to a range of conditions, including completion of an array of pre-commencement studies and preparation of management plans. The pre-commencement conditions relating to the Project's water supply and water management will require a significant amount of technical work, stakeholder consultation and administrative effort.<br>• Political risks and opportunities relate to possible changes to the land tenure system, permitting requirements and royalty or taxation regimes. The risk of adverse changes being imposed by the Federal or New South Wales State Governments are considered low.<br>• The systems for granting land tenure and issuing permits for developing and operating mining and minerals processing plants are well established in New South Wales. With a granted Development Consent, development of the Project is subject only to financing and conversion of the Mining Lease Applications to Mining Leases. The Project site is in an area of no particular environmental significance and the Project has the support of Local and State Government leaders.<br>• Levels of royalty and taxation, and methods for their calculation, are also well established. The political climate in Australia at present is focused on accelerating development of the regional and rural areas where the economy is perceived to have suffered in recent years. |
| Commercial | • Commercial risks and opportunities relate to achieving the forecast product sales volumes and metals prices, as well as to the certainty of supply and price of major Project operating inputs.<br>• In order to mitigate the risk with respect to metal market movements, the Company commissioned an independent report on the nickel and cobalt markets by CRU Strategies (UK) and has consulted extensively with producers, consumers and traders in both metals.<br>• The strategy for minimising the risk of sales shortfall is to establish a relationship with a few off-take partner(s) who will off-take a large portion (80% - 100%) of the product. It should be noted that cobalt supply is largely from Africa and that the projected nickel laterite developments tend to fall off rapidly in cobalt production capacity within the first five years of operation. |

| Risk/ Opportunity | Summary |
|---|---|
| Implementation | • Implementation issues include ensuring that the required technical standards in design and construction and that the Project is brought into production on or ahead of schedule. In general, the risk relates to failure of management to establish the optimum contracting strategy and structure and to monitor and control design and construction consultants and contractors to ensure that the Project is built to specifications, on time and within budget.<br>• An owner's team with a high level of expertise in process and engineering design, project controls and contracts establishment and administration, will be assembled. The team will have sufficient resources to establish and maintain a high level of monitoring and control. The personnel will be given responsibility levels sufficient to ensure optimal results. |
| Operations | • Operational issues which may constitute risks or opportunities include the recruitment and training of the required numbers of suitably qualified and experienced management, supervision and operations personnel, establishment of a suitable industrial relations structure for operations staff, awarding and administration of a maintenance contract, and ensuring that consumables and other supplies meet required specifications.<br>• The workforce establishment strategy pivots on the recruitment of an experienced operations management team and aggressive exploitation of the knowledge and experience gained from similar lateritic nickel operations currently operating. Early operator and maintenance personnel training programs are to be adapted for local employees and more experienced personnel from similar operations. |

## 1.11 Conclusions & recommendations

Due to the positive economics, SRK recommends that consideration be given to advancing the relevant aspects of the project to Feasibility Study level.

SRK's specific recommendations are as follows:

- Pit designs were not prepared as a basis for this Report. This Report uses the 2005 pit designs as it was considered that updated pit optimisation is likely to be similar to the results of the 2005 Study. SRK recommends that updated pit optimisations and pit designs be undertaken in the further study work to confirm the pit staging and to optimise the Mineral Reserve, ore and waste schedules. SRK recommends that updated pit optimisation be undertaken in the further study work to confirm the pit staging, limestone and waste schedules.
- While the current Project is intended to have a reduced water demand, supplementary water is likely to be required. Alternative sources of supply may need to be considered if the appropriate approvals are not obtained in the future. SRK recommends that Clean TeQ finalises the water the water balance as part of the FS and secures any additional water requirement.
- The pipeline operator and gas providers advised that insufficient gas availability will not be a risk to the Project, but that pricing is less certain and there is a risk of escalating prices. SRK recommends plans to secure gas supply contract be progressed.
- SRK recommends that landholding agreements are finalised as soon as practical.
- SRK recommends that all capital and operating cost estimates are reviewed and updated as required as part of the FS.
- Prior to completion of the FS, it is recommended that piloting the RIP system and downstream purification on fresh ore is carried out in order to produce samples and to confirm all process inputs for the full-scale plant.

## 1.12 Other relevant information

This Report focuses on the extraction of nickel and cobalt and does not focus on the extraction of scandium. SRK is aware that Clean TeQ is progressing additional work to assess the technical and economic feasibility of scandium production.

Where appropriate, SRK makes reference to scandium for context, but has not attributed any value to scandium in this assessment.

In a scandium context, the development of the HPAL methodology for the extraction of scandium is widely accepted. Metallica Minerals completed a PFS based on the HPAL process for scandium extraction, which included extensive metallurgical testwork. The Owendale (Platina), Flemington (Australian Mines) and the Nyngan (Scandium International Mining Corporation) projects have all undertaken testwork programs and studies, including metallurgical testwork validation of the HPAL process for scandium extraction, with recoveries similar to those achieved for nickel and cobalt.

The metallurgical testwork completed in the previous two feasibility studies on the Project typically tested scandium, as well as nickel and cobalt. All historical testwork confirms that scandium extraction using the HPAL process ranges from 80% to 90%, and is sometimes higher.

# 2 Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) has worked in conjunction with Clean TeQ Holdings Limited (Clean TeQ) and Widenbar & Associates and Mine & Development Services (Widenbar) to prepare this Technical Report on the Syerston Nickel Cobalt Project (the Project). The Technical Report, prepared in accordance with the Canadian National Instrument NI 43-101, validates the viability of mining and processing mineralisation at the Project.

The Syerston deposit is situated in central New South Wales, about 350 km WNW of Sydney. The Project is well supported by major centres, with the mining communities of Parkes, Dubbo and Condobolin all located within 100 km of the Project area.

Clean TeQ Holdings Limited, through its operating company Clean TeQ Limited (Clean TeQ), is an Australian-based ASX-listed company (ASX: CLQ) and a world leader in ion exchange technologies for materials processing applications, particularly in the mining sector through the use of a portfolio of proprietary technologies. Founded in 1990, Clean TeQ has invested heavily in research and development to produce commercial solutions to meet the future demands of a rapidly changing world that operates with limited resources.

## 2.1 Scope of work

The scope of work was to review and verify the technical work undertaken on the Project to sufficient detail to enable SRK and Widenbar to act as Qualified Persons and preparing documentation as required for the NI 43-101 compliant Technical Report on the Syerston Nickel Cobalt Project, in compliance with NI 43-101 and Form 43-101 F1 guidelines.

## 2.2 Work program

The Mineral Resource and Mineral Reserve Statement reported herein was prepared in conformity with generally accepted Canadian Institute of Mining (CIM) “Exploration Best Practices” and “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. This Technical Report was prepared following the guidelines of the Canadian Securities Administrators National Instrument 43-101 and Form 43-101 F1.

The Technical Report was assembled in Melbourne during the months of May and August 2017.

## 2.3 Terms of reference and purpose of the Technical Report

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) by SRK Consulting (Australasia) Pty Ltd (SRK), with Widenbar & Associates Pty Ltd and Mine & Development Services (Kitto) and Clean TeQ Holdings Limited (Clean TeQ) (the Owner) on the Syerston Nickel Cobalt Project (the Project), New South Wales, Australia.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK’s services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by the owners, subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits the owners to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party’s sole risk. The responsibility for this disclosure remains with the issuing companies. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report provides Mineral Resources, and a classification of resources prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

## 2.4 Qualifications of SRK and SRK's team

The compilation of this Technical Report was completed by the lead author, **Peter Fairfield**, Principal Consultant (Project Evaluation), BEng (Mining), FAusIMM (No 106754), CP (Mining). By virtue of his education, membership of a recognised professional association and relevant work experience, Peter Fairfield is an independent Qualified Person (QP) as this term is defined by the NI 43-101.

**Danny Kentwell,** Principal Consultant (Resource Evaluation), MSc Mathematics and Planning (Geostatistics), FAusIMM, conducted a review of the Mineral Resources and geological aspects of this Technical Report. Danny Kentwell, by virtue of his education, membership of a recognised professional association and relevant work experience, is an independent QP as this term is defined by NI 43-101.

The 2017 Mineral Resource Estimation was completed by **Lynn Widenbar**, Principal Consultant (Widenbar & Associates), BSc (Geology)(Hons), MSc (Mineral Exploration) (Hons), DIC, MAusIMM, MAIG. By virtue of his education, membership of a recognised professional association and relevant work experience, Lynn Widenbar is an independent Qualified Person (QP) as this term is defined by the NI 43-101.

The 2017 Mineral Resource Estimation was completed by **Peter Kitto**, Principal Consultant (Development and Mining Services), BSc (Geology)(Hons), FAusIMM. By virtue of his education, membership of a recognised professional association and relevant work experience, Peter Kitto is an independent Qualified Person (QP) as this term is defined by the NI 43-101.

**Simon Walsh,** SRK Associate Principal Metallurgist, BSc (Extractive Metallurgy & Chemistry), MBA Hons, MAusIMM(CP), GAICD, undertook a review of the metallurgical, mineral processing and infrastructure aspects of the Project. By virtue of his education, membership of a recognised professional association and relevant work experience, Simon Walsh is an independent QP as this term is defined by NI 43-101.

**Anthony Stepcich**, Principal Consultant (Mining), BEng (Mining), MSc (Mineral Economics), GDip (Finance & Investment), Dip (Technical Analysis), FAusIMM CP(Mining), conducted a peer review of the non-geological aspects of this Technical Report. Anthony is, by virtue of her education, membership of a recognised professional association and relevant work experience, an independent QP as this term is defined by NI 43-101.

Table 2-1 defines the areas of responsibility for the QPs, who all meet the requirements of independence as defined in NI 43-101.

## 2.5 Declaration

SRK's opinion contained herein, effective 30 October 2017 is based on the 2016 Syerston Nickel Cobalt Pre-Feasibility Study Report and information collected by SRK throughout the course of SRK's investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This Report may include technical information that requires subsequent calculations to derive subtotals, totals and weighted averages.

Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where errors occur, SRK does not consider them to be material.

SRK/ Widenbar are not an insider, associate or affiliate of Clean TeQ, and neither SRK nor any affiliate has acted as advisor to Clean TeQ, its subsidiaries or its affiliates in connection with this Project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

The SRK Group comprises over 1,400 professionals, offering expertise in a wide range of resource and engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and its ownership rests solely with its staff. This permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports, and independent feasibility evaluations to bankable standards, on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

**Table 2-1: List of qualified persons**

| Position | Employer | Last site visit date | Professional designations | Area of responsibility and Report sections |
|---|---|---|---|---|
| Peter Fairfield<br>Principal Consultant<br>(Project Evaluations) | SRK | 07/06/17 | BEng (Mining),<br>FAusIMM<br>CP (Mining) | Sections 1 - 6,<br>Sections 15 - 16,<br>Sections 18 - 27 |
| Danny Kentwell<br>Principal Consultant<br>(Resource Evaluation) | SRK | NA | MSc Mathematics &<br>Planning (Geostatistics),<br>FAusIMM | Sections 7 - 12,<br>Section 14 |
| Simon Walsh<br>Principal Metallurgist | Simulus Engineers (SRK Associate) | NA | BSc (Extractive Metallurgy),<br>MBA Hons,<br>F AusIMM (CP),<br>GAICD | Section 13<br>Section 17<br>Section 18 |
| Tony Stepcich<br>Principal Consultant<br>(Mining) | SRK | NA | BEng (Mining)<br>MSc (Mineral Economics)<br>G Dip (Finance & Investment)<br>Dip (Technical Analysis)<br>FAusIMM(CP) | SRK Peer Review |
| Lynn Widenbar<br>(2017 Mineral Resource Estimation) | Widenbar & Associates | 21/09/17 | BSc (Geology),<br>MSc (Mineral Exploration)<br>DIC<br>MAusIMM<br>MAIG | Sections 7.2.6<br>Section 11.8<br>Section 11.9<br>Section 12.4<br>Section 14.2 |
| Peter Kitto<br>(2017 Mineral Resource Estimation) | Development and Mining Services | NA | BSc (Geology),<br>FAusIMM | Sections 7.2.6<br>Section 11.8<br>Section 11.9<br>Section 12.4<br>Section 14.2 |

# 3 Reliance on Other Experts

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

## 3.1 Mineral tenure

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon information sourced from regulatory authorities.  This information is used in Section 4 of the Report.

## 3.2 Markets

THE QP's have not independently reviewed the markets, metal price forecast information.  The QPs have relied upon the marketing and market conditions for nickel and cobalt, which have been provided in a number of reports and sources supplied by Clean TeQ in the 2016 PFS.  These are presented in Section 19 and 22.

# 4 Property Description and Location

## 4.1 Property location

The Syerston deposit is situated in central New South Wales, about 350 km WNW of Sydney. The Project is well supported by major centres, with the mining communities of Parkes, Dubbo and Condobolin all located within 100 km of the Project area. The town of Fifield is located 4 km from the Project area. The Project area experiences a subtropical, dry type of climate, with very low rainfall, high daytime temperatures in summer and low minimum temperatures in winter.

The district is predominantly used for agriculture, with crops including wheat, barley and oats. Grazing of sheep and cattle is also common throughout the district. Due to widespread clearing for agriculture over the last 100 years, very little of the original vegetation remains.

The Project area is located on three pastoral properties and includes previously mined land (magnesite), State Forest and Crown Land. The Fifield State Forest occupies a small part of the Project area situated along the northern border, and the Unoccupied Crown Land is found in the northeastern corner of the Project area. Neither of these areas are within the relevant development area.

One of the Project's competitive advantages is its proximity to existing infrastructure. The Project is located close to the Moomba–Sydney natural gas pipeline, and a rail line within 20 km of Syerston and bitumen roads provide good access to the site. The major centres have excellent infrastructure including transport, airport and rail facilities, all of which are available to service the requirements of the Project. The Project and associated infrastructure are located in the shires of Lachlan and Parkes, while the borefield supplying water for the Project is located in the Forbes Shire.

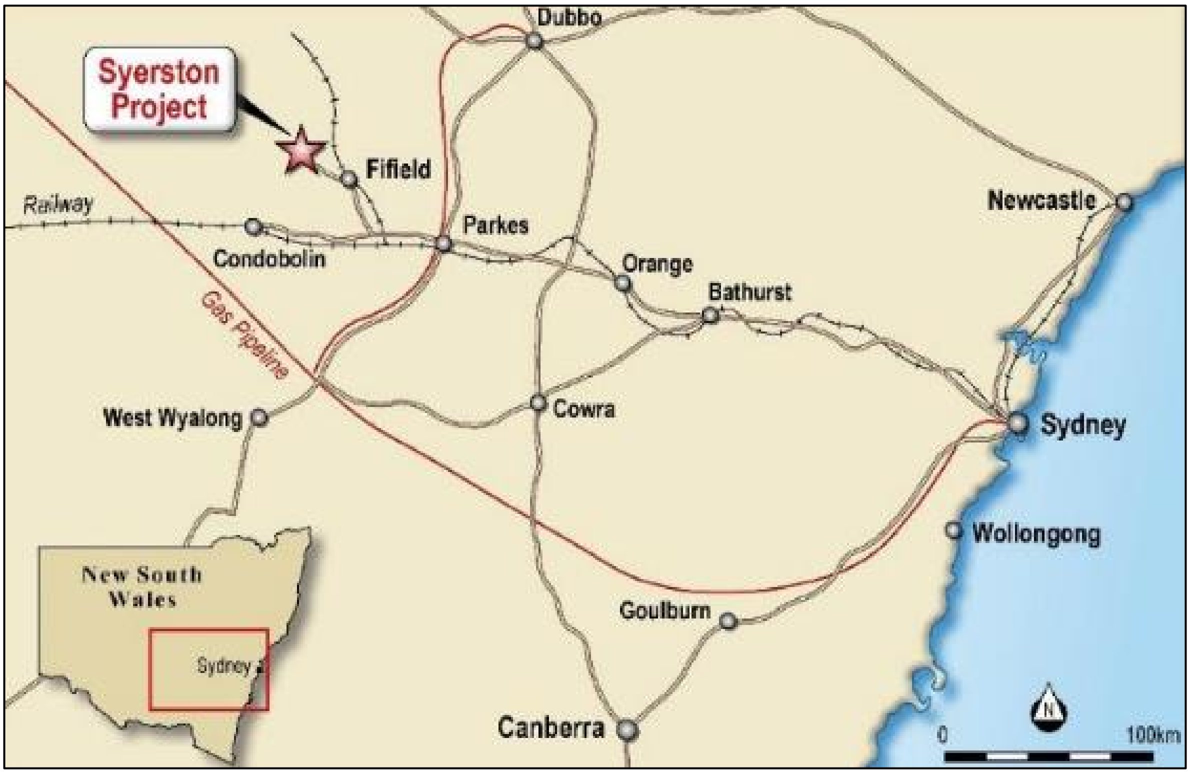



**Figure 4-1: Project location plan**

## 4.2 Land tenure

The Exploration Licence (EL) and Mining Lease Applications (MLAs) for the Project are shown in Figure 4-2.

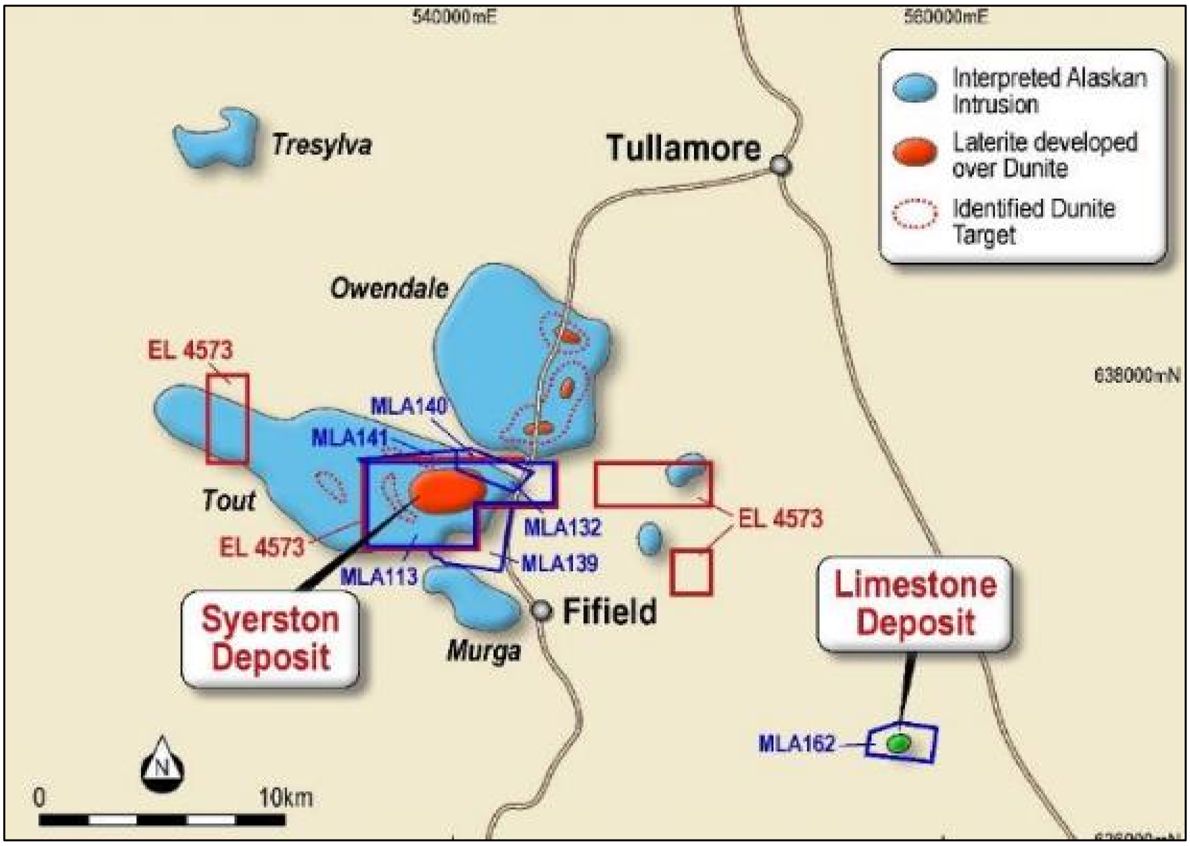



**Figure 4-2: Tenement ownership overview**

### 4.2.1 EL 4573

Exploration Licences are granted and administered by the New South Wales Department of Mineral Resources under the *Mining Act 1992 and Regulations*. The holder of an Exploration Licence has a priority right to apply for one or more Mining Leases over the area of the Exploration Licence.

Exploration Licences are applied for and granted under the graticular system whereby the State is divided into graticular sections called blocks and units. A unit is the basic graticular section of one minute of latitude by one minute of longitude. A block is comprised of a section of five minutes of latitude by five minutes of longitude, i.e. 25 units.

Scandium21 owns EL 4573 covering the area of the Project and facilities. The status of the Exploration Licence as at March 2015, is listed in Table 4-1.

**Table 4-1: Exploration licence details**

| Licence No. | Date of Grant | Expiry Date | Blocks Granted | Ownership | Annual Expenditure (AUD) |
|---|---|---|---|---|---|
| EL 4573 | 17/08/1993 | 16/08/2018 | 19 (4 areas) | 100% | 49,000 |

EL 4573 has been granted subject to the provisions of the *Mining Act 1992* and to the conditions of the licence. The licence holder has exclusive rights to prospect in the exploration area for Group 1 minerals. Group 1 minerals are elemental minerals (metallics) and rare earth minerals (including scandium), nickel, cobalt and platinum.

At the Expiry Date, Clean TeQ has the option to renew the Exploration Licence for an additional period of three years, subject to the licence conditions being met. To date, all Exploration Licence conditions for EL 4573 have been met.

### 4.2.2 Mining lease applications

The boundaries of the MLAs over the Project area and limestone quarry are shown in Figure 4-3 and Figure 4-4. The status of each MLA is shown in Table 4-2.

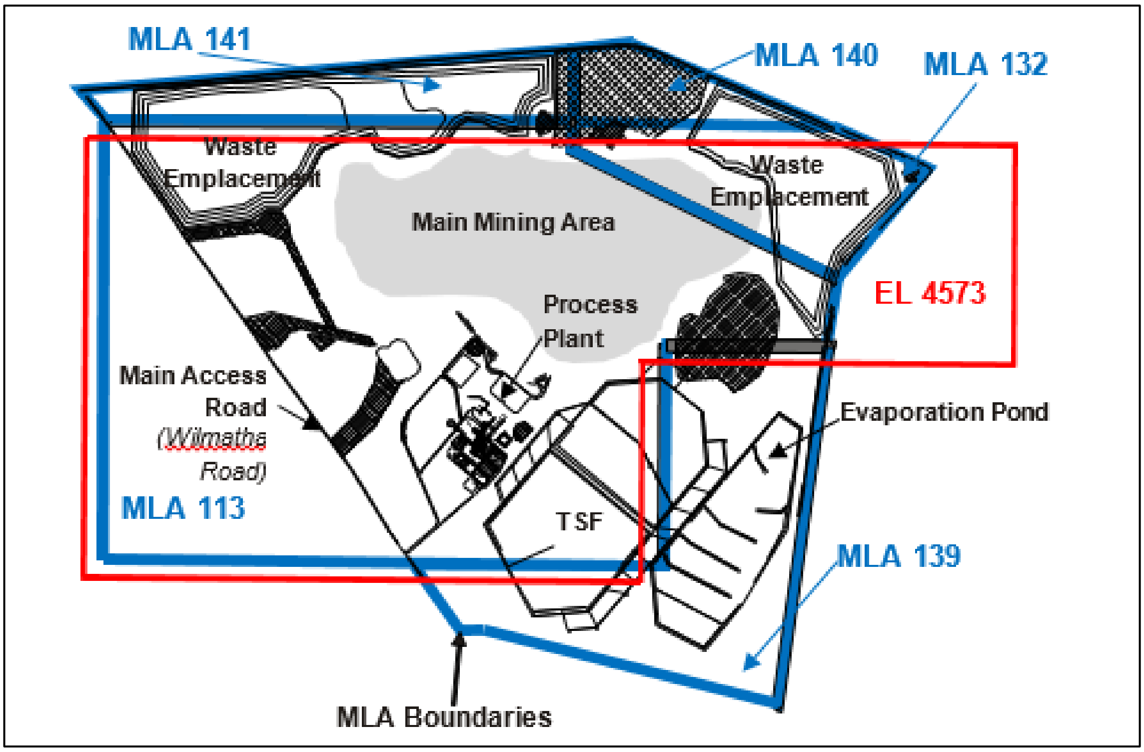



**Figure 4-3: Mining lease application boundaries – main project area**

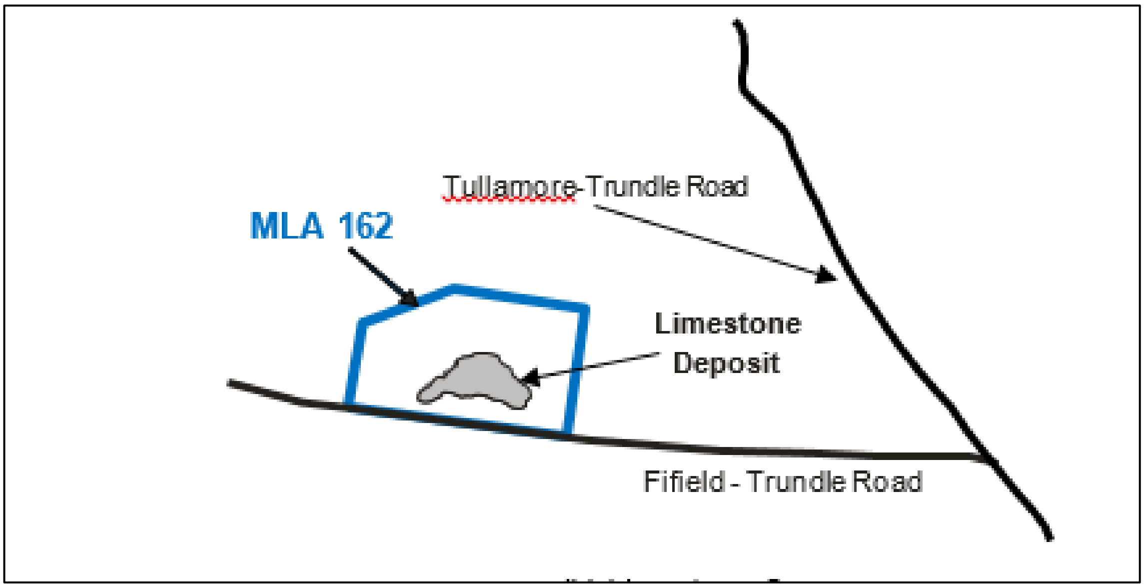



**Figure 4-4: Mining lease application boundaries – limestone quarry**

**Table 4-2: Status of mining lease applications**

| Licence Application No. | Area | Application date | Grant status (25/7/2017) | Interest |
|---|---|---|---|---|
| MLA 113 | 8 units | 10 August 1998 | Pending | 100% |
| MLA 132 | 200.00 ha | 20 September 1999 | Pending | 100% |
| MLA 141 | 137.5534 ha | 10 December 1999 | Pending | 100% |
| MLA 140 | 77.7845 ha | 10 December 1999 | Pending | 100% |
| MLA 139 | 421.0488 ha | 10 December 1999 | Pending | 100% |
| MLA 162 | 390.00 ha | 27 September 2000 | Pending | 100% |

Mining Lease Applications have been made for the mine, processing and associated infrastructure in accordance with the *Mining Act 1992 and Regulations*. The *Mining Act* is administered by the Department of Mineral Resources and allows the grant of a Mining Lease for any mineral or minerals defined under the *Mining Act*. It also prescribes a royalty rate payable on all minerals recovered from any Mining Lease.

## 4.3 Landholding

Figure 4-5 and Figure 4-6 show the landholding by Scandium21 over the Project area and limestone quarry.

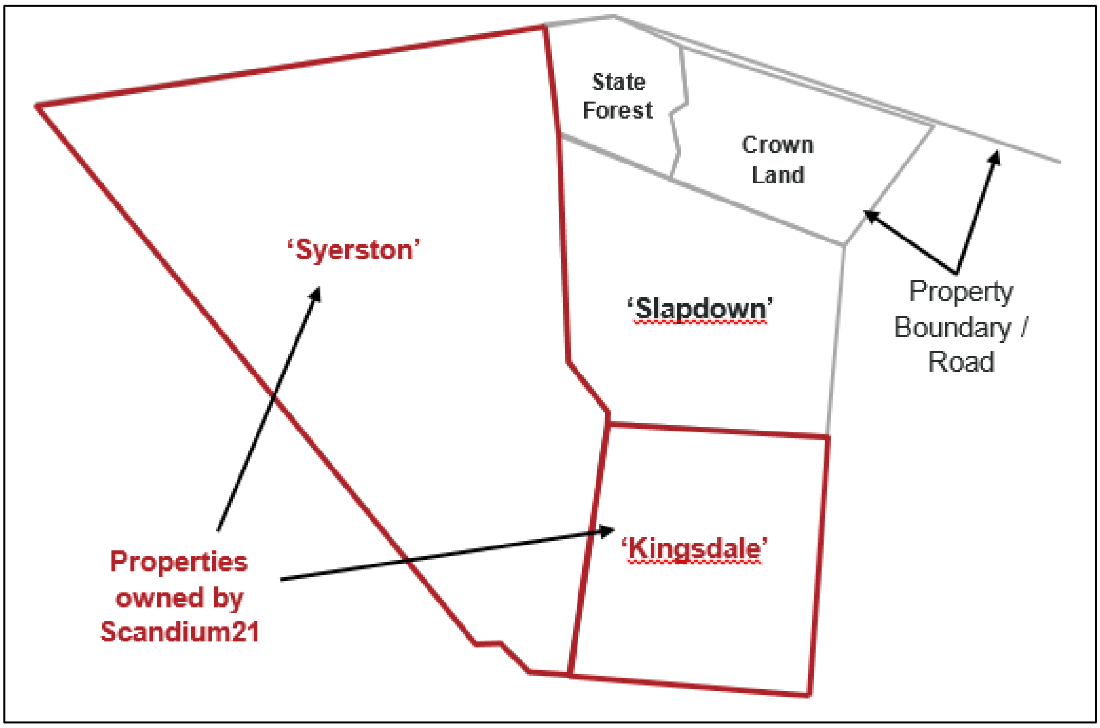



**Figure 4-5: Scandium21's landholding – Main project area**

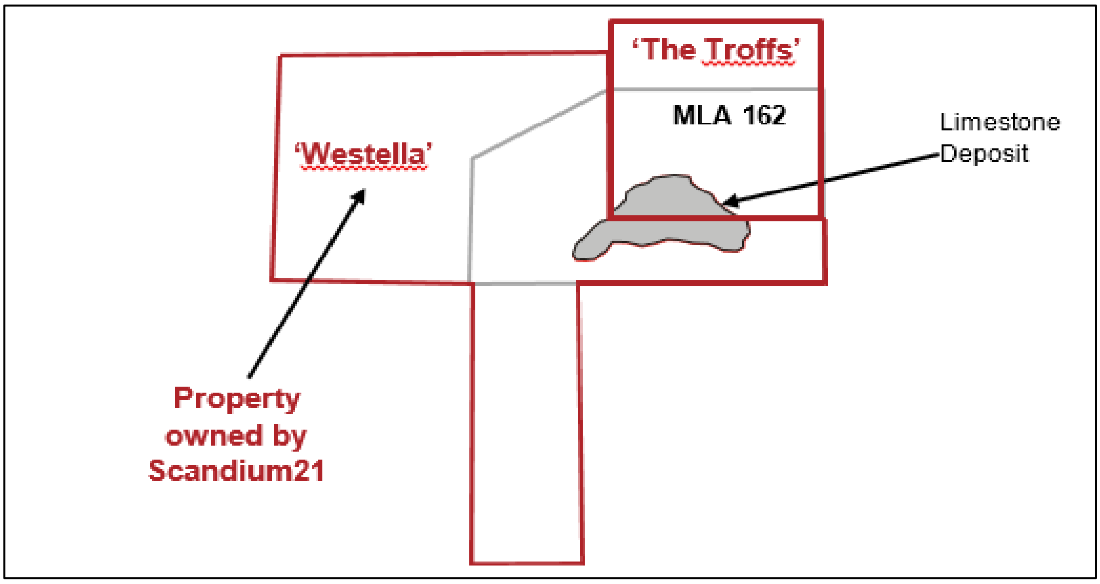



**Figure 4-6: Scandium21 landholding – limestone quarry**

Scandium21 also owns the Kelvin Grove property to the northeast of the Project area. All properties are currently tenanted. Agreements to acquire or option the Slapdown, The Troffs and rail siding properties are currently being negotiated. Negotiations for new option, acquisition or access/ easement agreements will need to be undertaken for part of the gas pipeline and Fifield bypass.

## 4.4 Development consent

An EIS was prepared following extensive consultation and assessment studies in late 2000 as a requirement to apply for Development Consent for the Project. The Project was granted Development Consent under the *Environmental Protection and Assessment Act* in May 2001.

In 2005, Ivanplats submitted an application to the Department of Infrastructure, Planning and Natural Resources (DIPNR) to modify the Development Consent. The application relates to an increase in autoclave feed rate from 2.3 Mtpa to 2.5 Mtpa, the removal of the refinery, and modifications to the timing of some of the plans, studies etc. required as conditions of the Development Consent. Ivanplats prepared a Statement of Environmental Effects (SEE) for the modification application. The Notice of Modification was approved by the NSW State Government on 15 October 2005, as requested in the SEE.

The grant of a Mining Lease over freehold land requires, for practical purposes, the consent of the landholder. To ensure that obtaining such consent will not delay grant of the Mining Leases, Ivanplats (now Scandium21) purchased the majority of the land over the Project area, which provided a significant advantage for the approval of the Mining Leases. At the time of the application, all landholders provided consent for the MLAs.

Having completed an EIS and having a Development Consent in place offers a significant advantage over other scandium projects, due to the time and relative costs to complete these activities.

The Project was also referred under the *Commonwealth Environmental Protection and Biodiversity Conservation Act* and it was deemed that the Project is not a controlled action under this Act.

In May 2016, a modification application was submitted to the Department of Planning for the inclusion of scandium oxide as a product, as well as nickel and cobalt sulphates as alternatives to the sulphide products currently in the approved Development Consent. The modification sought to update the road upgrades and community enhancement plans to reflect contemporary requirements.

The approval of the Notice of Modification was granted on 12 May 2017.

## 4.5 Royalties

Royalties are payable to the NSW State Government and Ivanhoe Mines at the rate of 4.00% and 2.5% (after State Government royalty) respectively.

The 4% NSW State Government mineral royalty has been applied over the life of mine. The State Government royalty is 4% of the net revenue. Net revenue is gross revenue less “allowable deductions”. Details provided by the NSW Government are available from:

http://www.resourcesandenergy.nsw.gov.au/miners-and-explorers/enforcement/royalties/royalty-rates

Broadly, allowable deductions include operating costs incurred after ore is delivered to the ROM pad. It therefore excludes all mining costs. Depreciation of allowable deduction is restricted to the proportion of the assets’ utilisation that contributes directly to upgrading the mineral from the first stockpile to disposal. All assets (except tailings dams) are to be depreciated using the reducing balance method at a rate of 11.25% per annum. Tailings dams are depreciated on a straight-line basis over the useful life of the dam. The allowable deduction for administration is capped at 33% of total administration cost, as shown in the following formula:

Royalty = (Revenue - Processing Cost – (33% x Site Administration Cost) – Allowable Depreciation Deduction - Transport Costs) x 4%

The 2.5% royalty payable to Ivanhoe Mines has been applied over the life of the mine. The royalty is calculated as follows:

Ivanhoe Mines Royalty = (Revenue – NSW Royalty) x 2.5%

## 4.6 Taxes

A 27.5% company tax rate has been applied.

# 5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

## 5.1 Accessibility

The Project is well supported by major centres; with the mining community of Parkes (population of 10,000) located 80 km to the southeast; Dubbo (population of 32,000) located 120 km to the northeast; and Condobolin (population of 3,700) located 60 km to the southwest. Other towns of noticeable size in the vicinity of the project include Tullamore (400), Tottenham (380), and Trundle (550).

The Project is 4 km to the north of the small township of Fifield. Fifield has an extensive history of mining for platinum and magnesite over the last 100 years, with magnesite mining ceasing during the mid-1980s.

## 5.2 Land use

The district is predominantly used for agriculture, with crops within the region including wheat, barley and oats. Grazing of sheep is also common throughout the district. Due to widespread clearing for agriculture over the last 100 years, very little of the original vegetation remains.

The Project area is located on three pastoral properties, previously mined land (magnesite), State Forest and Crown Land. Farming is focused on cropping, and the area is also used for sheep and cattle grazing. The Fifield State Forest occupies a small part of the Project area, along the northern border, and the Unoccupied Crown Land is in the northeastern corner of the Project area.

There are scattered remnant native trees in the wheat paddocks, occurring singly or as small clusters. The hillier sites on the farmland retain a greater coverage of native vegetation, but in general, the vegetation has been significantly thinned in the past to promote growth of grasses for grazing. Dense regeneration of White Cypress Pine has occurred on some of these areas. Strips of natural vegetation have been left along the main watercourses in the farmed areas to prevent soil erosion during periods of heavy rainfall.

## 5.3 Topography

Much of the northeastern quarter of the lease area contains land that has been subject to open cut mining. The mining areas have been partly rehabilitated, but several open pits remain, some of which contain water. There are the remnants of the old mining infrastructure scattered throughout this part of the area. Fifield State Forest has been extensively logged in the past, with a few mature trees in this part of the Project area. There is a belt of trees through the centre of the area that is associated with low-lying land and indefinite watercourses. Some of this area of trees has been cleared in recent years and the land used for cropping.

The Project area occurs on gently undulating country ranging from 270 m to 320 m in altitude. A shallow watercourse bisects the area, running diagonally across the Project to the northeast and forming two parallel tributaries in the southwest. Several low gravelly hills occur across the site with broad shallow valleys between.

## 5.4 Climate

The Project area experiences a subtropical, dry type of climate, i.e. a very low rainfall, high daytime temperatures in summer and low winter minimum temperatures; approaching the warm temperature type – with no noticeable winter and year-round rainfall.

Climatological data from Condobolin Agricultural Research Station shows an annual mean daily maximum temperature of 24.5°C, a mean daily minimum of 10.2°C, and a mean annual rainfall of 458 mm (as shown in Table 5-1 and Figure 5-1). The highest maximum temperatures are recorded during January (46°C), and the lowest during July (-7°C).

Rainfall is evenly distributed through the year. The winter months have the highest soil moisture levels due to the lower evaporation.

**Table 5-1: Climate data (Condobolin agricultural research station)**

| | Jan | Feb | Mar | Apr | May | Jun | Jul | Aug | Sep | Oct | Nov | Dec | Annual Average |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mean maximum temperature (°C) | 33.8 | 32.8 | 29.4 | 24.6 | 19.6 | 15.7 | 15 | 16.9 | 20.6 | 25 | 28.8 | 31.7 | 24.5 |
| Mean minimum temperature (°C) | 18.2 | 18.2 | 14.8 | 9.9 | 6.5 | 3.9 | 2.8 | 3.5 | 5.8 | 9.4 | 13.1 | 15.7 | 10.2 |
| Mean rainfall (mm) | 46.9 | 44.8 | 39.6 | 30.6 | 35.7 | 31.3 | 35.9 | 33.7 | 31.4 | 47.4 | 38.6 | 41.6 | 457.8 |

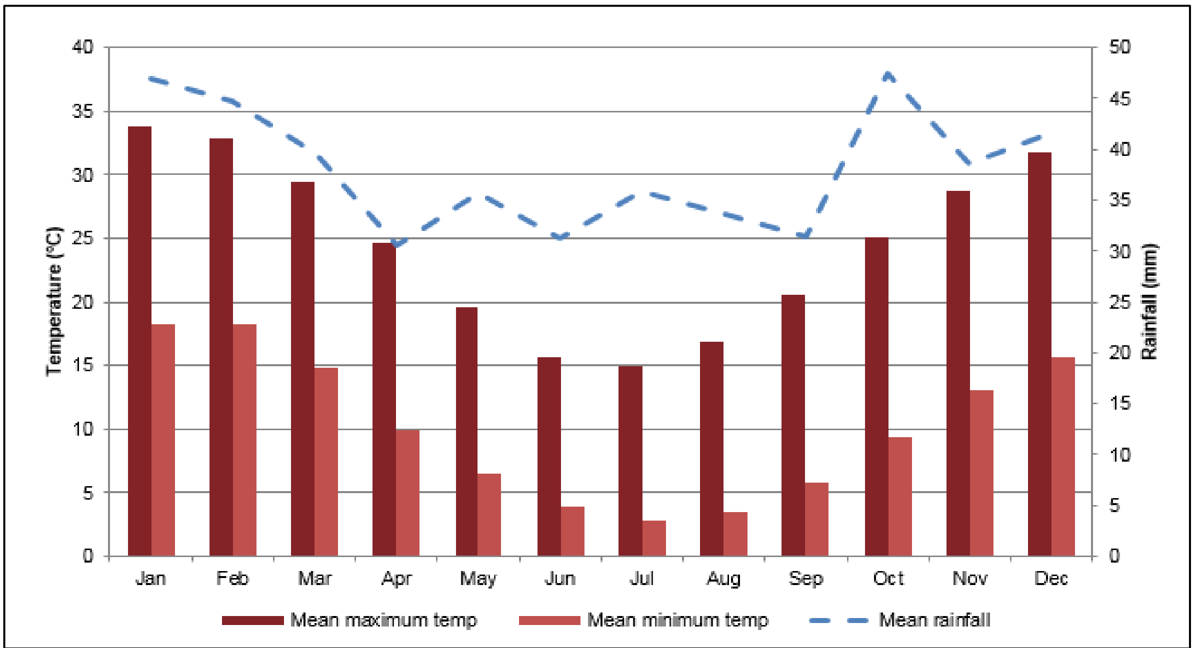



**Figure 5-1: Climate data from the Condobolin agricultural research station**

## 5.5 Infrastructure and local resources

Syerston's competitive advantage in part is based on its proximity to existing infrastructure. The Project is located in close proximity to the Moomba–Sydney natural gas pipeline, and has access to competitive gas/ electricity prices. A rail line lies within 20 km of Syerston, giving it access to the ports of Sydney and Newcastle. Bitumen roads provide good access to the site.

The major centres have excellent infrastructure, including transport, airport and rail facilities – all of which are available to service the requirements of the Project. In addition, the mining industry in the region is well understood and supported by the major centres. There are several large operating mines in the region, including the Northparkes, Peak Hill, Browns Creek, Lake Cowal and Cadia mines.

The Project and associated infrastructure is located within the Lachlan and Parkes shires, with the borefield located in the Forbes Shire. All parties have responded positively to the re-initiation of the Project.

Mining activity in the region supports the economies of the local towns. In particular, the former magnesite mine, which forms part of the Project area, employed up to 60 local residents. Tin, copper and gold have also been mined to varying degrees in the district.

# 6 History

The Project is located within the Fifield District of central New South Wales. The district, known for its unique geological anomalism, is the only recorded source of platinum production in Australia. Reported production from the district totalled 639.5 kg of platinum between 1887 and the mid-1960s. The platinum was derived from deep leads draining areas of largely concealed ultramafic intrusions. Minor gold production has also been reported from the district.

Due to the occurrence of platinum within deep leads, the district became the focus of renewed interest in primary platinum. Recorded regional exploration commenced in the 1960s when several companies, including Anaconda Australia Limited, CRA Exploration and the Shell Company of Australia Limited, explored the area for base metals and platinum group metals (PGMs) mineralisation until the 1980s. The work included geophysical surveys, evaluation of the deep leads and exploratory drilling.

EL 2663 and EL 2664 were granted to Freshwater Resources and Balmoral Resources in September 1986 for a period of two years. The tenements were successfully vended into the Noble Resources NL (Noble) public float in December 1986.

## 6.1 Ownership

### 6.1.1 Noble Resources

Noble conducted exploration until March 1988 primarily searching for primary platinum within the Tout Complex. The exploration failed to locate or identify any prospective platinum targets, although it did identify the occurrence of a large laterite profile over the dunite portion of the intrusive enriched with nickel, cobalt and platinum. During 1988, Poseidon Limited entered into a joint venture (JV) agreement with Noble. Poseidon was entitled to earn 51% of the project by spending AUD1.2 million over a period of five years.

Poseidon managed exploration and primarily focused on the delineation of the laterite mineralisation by systematic rotary air blast (RAB) drilling, followed by metallurgical testwork. In December 1992, Poseidon Limited withdrew from the JV, relinquished its 51% ownership of the Project and returned management to Noble.

Noble's focus during 1992 reverted to exploring for platinum mineralisation within the bedrock. During this period, two diamond holes were drilled at the northern end of the dunite intrusive, but these failed to intersect economic platinum grades.

Between 1993 and 1996, further evaluation of the laterite was undertaken for nickel and cobalt, including further RAB/ aircore drilling and metallurgical investigations. The original EL 2663 and EL 2664 expired during 1992 and a new application was submitted for the same ground in a single tenement. EL 4573 was granted to Noble in August 1993.

During 1997, further evaluation of the laterite was advanced by the completion of 341 reverse circulation (RC) holes and three PQ triple tube diamond holes. This led to the resource being remodelled and estimated by EMC consultants, which formed the basis of a PFS. During this time, the Company entered into a JV agreement and option deed with Uranium Australia, later renamed Black Range Minerals (Black Range), to provide further funding for the Project.

Commencing March 1998, Fluor Daniel Pty Ltd conducted a PFS on the nickel/ cobalt resource and the mining, processing and infrastructure facilities required for the development of the deposit. The Fluor Daniel PFS was successfully completed in August 1998, indicated the Project's viability and recommended the Project be progressed to the bankable feasibility stage. During this period, pursuant to the option deed, Black Range acquired all outstanding shares in the holding company for the deposit, effectively taking control.

### 6.1.2 Black Range Minerals

During this period, the Company undertook high level investigations on the extraction and recovery of nickel and cobalt from the Project.

During 1998, the Feasibility Study for the Project commenced with the establishment of the Feasibility Project Team. During this initial period, a number of key opportunities for project enhancement were identified.

During 1999, an extensive drilling program was completed as the basis for the feasibility study. This included completion of 720 RC holes, seven PQ triple tube diamond holes and nine large diameter bucket holes.

SNC-Lavalin was commissioned by Black Range in June 1999 to undertake a Feasibility Study for the Project, which was completed in June 2000.

Based on the Feasibility Study, Black Range completed an EIS as a basis for its Development Consent application. The EIS was submitted to the NSW Government in November 2000 for assessment. Subsequently, in May 2001, the Project's Development Consent was granted.

### 6.1.3 Ivanplats Syerston

In 2004, Black Range was placed in voluntary administration. The Project, via Black Range Metals Syerston Pty Ltd, was acquired by Ivanplats Nickel & Platinum (INP). INP took control of the Company and Project in July 2004, and changed the company name from Black Range Metals Syerston Pty Ltd to Ivanplats Syerston Pty Ltd (Ivanplats).

Ivanplats commissioned SNC-Lavalin to update the Feasibility Study completed by Black Range; this was completed in 2005. A resource estimate of 107.1 Mt was established, grading at 0.65% nickel and 0.105% cobalt. The Project was evaluated over a 20-year timeframe. The plant design envisaged laterite ore processing using an HPAL environment with a 2.5 Mtpa autoclave feed capacity and mean recoveries of 93%. The processing plant had an output capacity of 25,000 t of nickel metal and 5,000 t of cobalt, with an average life of mine production rate of 17,800 tpa nickel and 2,100 tpa cobalt.

The overall capital cost was USD760M (AUD1.0Bn) with operating costs running at USD2.68/lb nickel, after cobalt credits. At the 2005 base case prices of US4.10/lb nickel and US12.00/lb cobalt, the Project generated an internal rate of return (IRR) of 5.6%, which was regarded as insufficient to justify development.

### 6.1.4 Clean TeQ

In November 2014, Clean TeQ entered into an agreement to acquire 100% of the shares in Ivanplats Syerston. The agreement was completed on 31 March 2015 giving Clean TeQ 100% ownership of Ivanplats Holding Company and all subsidiaries and assets, including the Syerston Project.

In 2016, Clean TeQ prepared a PFS as an update to the 2005 Study.

The 2016 PFS was based on a 2012 JORC Code compliant Measured, Indicated and Inferred resource, completed by McDonald Speijers Pty Ltd (McDonald Speijers). The resource estimate was based on more than 1,200 historical drill holes. Based on technical and cost inputs from historical studies, open pit modelling and production scheduling for the 2005 Feasibility Study were used. The resource has been updated under 2012 JORC Code guidelines. As there were only very minor changes as part of the update, the original 2004 resource and corresponding mine plan were adopted.

## 6.2 Outline of exploration (drilling) programs

The major phases of previous exploration are summarised in Table 6-1. The amounts of drilling indicated were based on holes in the available database, supplemented by information from some archived annual reports.

**Table 6-1: Exploration phases**

| Time period | Operator | Drilling type | Number of holes | Comments |
|---|---|---|---|---|
| Sep 1986 - Mar 1988 | Noble Resources | RAB | 590 | Locations of holes no longer known |
| Mar 1988 - Mar 1989 | Noble-Poseidon JV | RAB | 192 | Initially Pt exploration; lateritic Ni-Co potential recognised |
| Sep 1989 - Mar 1990 | Noble-Poseidon JV | RAB | 18 | Probably magnesite exploration; locations unknown |
| | | RAB | 10 | Initial metallurgical samples, mineralogy |
| Sep 1991 - Sept 1992 | Noble-Poseidon JV | DDH | 2 | Bedrock Pt targets |
| Aug 1993 - Aug 1994 | Noble Resources | RAB | 3 | |
| | | Aircore | 20 | |
| Aug 1995 - Aug 1996 | Uranium Australia | Aircore | 128 | Infill in lateritic Ni-Co resource |
| Aug 1997 - Aug 1998 | Uranium Australia | RC | 341 | Infill drilling |
| | | DDH | 5 | PQ3 core for density and metallurgical tests |
| Aug 1998 - Oct 2000 | Black Range Minerals | RC | 732 | Infill drilling, largely replacing aircore holes |
| | | DDH | 8 | PQ3 core mainly for density and metallurgical tests |
| | | Calweld | 9 | Large diameter holes for metallurgical bulk samples |
| Feb 2005 - Mar 2005 | Ivanplats Syerston | RC | 117 | Local infill drilling on 30 x 30 m pattern |
| Feb 2005 - Mar 2005 | Ivanplats Syerston | RC | 32 | Local infill to investigate higher grade Pt occurrences<br>Note: Results not available for use in resource modelling. |
| Feb 2005 - Mar 2005 | Ivanplats Syerston | RC | 26 | Twinning of some earlier RC and Calweld holes.<br>NOTE: Results not available for use in resource modelling |
| Aug 2014 | Ivanplats Syerston | RC | 14 | Testing for Sc over peripheral pyroxenites in the northern part of the Project area |
| Apr 2015 | Scandium21 | RC | 34 | Further testing for Sc over peripheral pyroxenites in the northern part of the Project area |
| Nov 2015 | Scandium21 | RC | 58 | Infill drilling in areas of elevated Sc |

## 6.3 Previous resource estimates

Of the numerous historical resource estimates since 1993, two are considered significant and comparable to the current resource. These are the 1999 resource estimate completed by SNC-Lavalin for Black Range Minerals and the 2005 resource completed by McDonald Speijers for Ivanplats Syerston Pty Ltd. Both estimates used NiEq cut-off grades.

**Table 6-2: 1999 resource estimate – SNC-Lavalin (JORC Code)**

| Category | Cut-off grade NiEq (%) | Inventory (Mt) | NiEq (%) | Nickel grade (% Ni) | Cobalt grade (% Co) |
|---|---|---|---|---|---|
| Measured | 0.65 | 79.3 | 1.16 | 0.72 | 0.12 |
| Indicated | 0.65 | 5.2 | 0.92 | 0.59 | 0.09 |
| Meas + Ind | 0.65 | 84.5 | 1.15 | 0.71 | 0.12 |
| Inferred | 0.65 | 11.5 | 0.97 | 0.53 | 0.12 |

The 1999 resource NiEq values were calculated as NiEq% = Ni% + (Co% x 3.64). The values were based on the following assumptions:

- Nickel price of USD2.75/lb
- Cobalt price of USD10.00/lb
- Metallurgical recoveries not considered.

**Table 6-3: 2005 Resource estimate – McDonald Speijers (NI 43-101 guidelines)**

| Cut-off NiEq (%) | Class | Inventory (Mt) | NiEq* (%) | Cont. Metal (NiEq kt) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|---|---|
| 0.6 | Measured | 51.8 | 1.06 | 549 | 0.73 | 379 | 0.111 | 58 |
| 0.6 | Indicated | 47.1 | 0.88 | 417 | 0.59 | 277 | 0.100 | 47 |
| 0.6 | Meas + Ind | 98.9 | 0.98 | 966 | 0.66 | 657 | 0.106 | 105 |
| 0.6 | Inferred | 8.2 | 0.84 | 69 | 0.54 | 45 | 0.098 | 8 |

Note:
* The 2005 resource NiEq values were calculated as NiEq% = Ni% + (Co% x 2.95), and used assumed metal prices of USD4.00/lb Ni and USD12/lb Co, and an USD/AUD exchange rate of 0.70.

The average overall metallurgical recoveries to final product were estimated to be 90.0% for Nickel and 88.9% for Cobalt.

The 2016 resource estimate undertaken by McDonald Speijers described in this Report was reported to the Australian Market under the guidelines of the JORC Code (2012) on 22 August 2016.

# 7 Geological Setting and Mineralisation

## 7.1 Regional geology

The regional basement consists of sediments of the Ordovician to Devonian Girilambone Group, which consists mainly of sandstones with interbedded siltstones, shales, local limestones and some volcanics.

These sediments have been intruded by a number of probably late Ordovician, Alaskan-type, mafic to ultramafic complexes that are typically zoned, with a core of ultramafic rock surrounded more or less concentrically by pyroxenite and then by gabbro to tonalite. These intrusions occur in a roughly north–south trending belt.

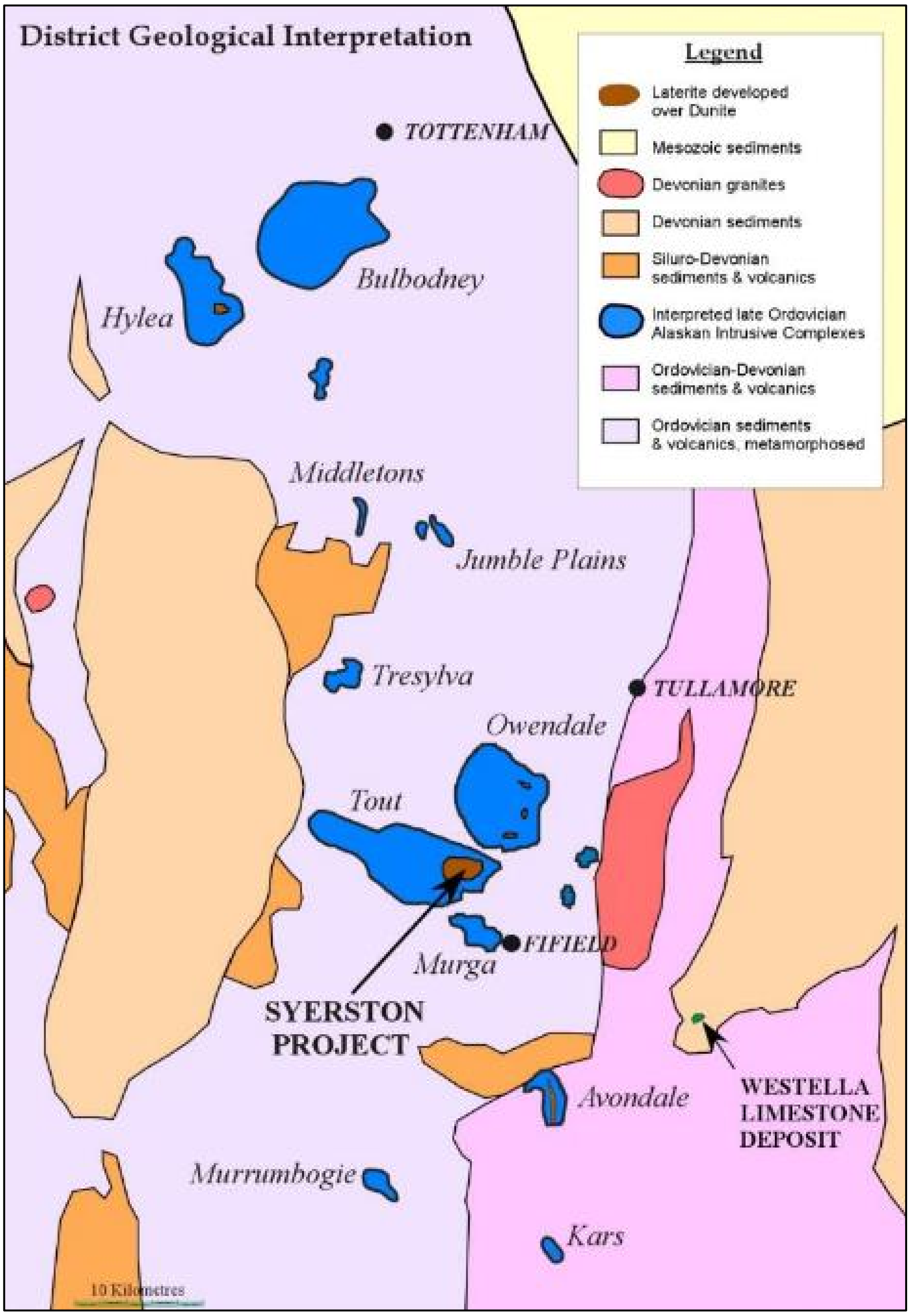



**Figure 7-1: District geological setting**

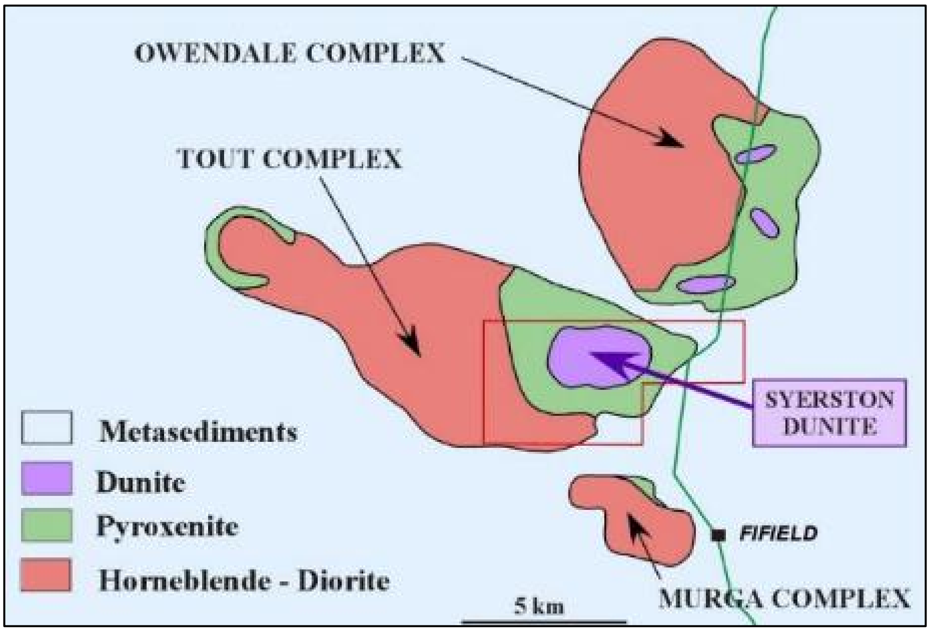



**Figure 7-2: Intrusive complexes in the Syerston area**

The region was affected by lateritisation during the Tertiary, which was followed by deposition of alluvial clays to gravels in the Tertiary drainage system as the climate became drier and the valleys silted up.

## 7.2 Local and property geology

The deposit lies over a mafic to ultramafic intrusive complex known as the Tout Complex (sometimes also referred to as the Flemington Intrusion). The lateritic weathering profile in which Ni, Co and Pt accumulated developed preferentially over the dunite core of the intrusion, which is about 4 km by 2 km in area (Figure 7-3, Figure 7-4, Figure 7-5).

Over the dunite, the lateritic profile can reach 35 - 40 m in thickness, but it thins markedly over the surrounding pyroxenites so that the base of the deposit has a basin-like form (Figure 7-4).

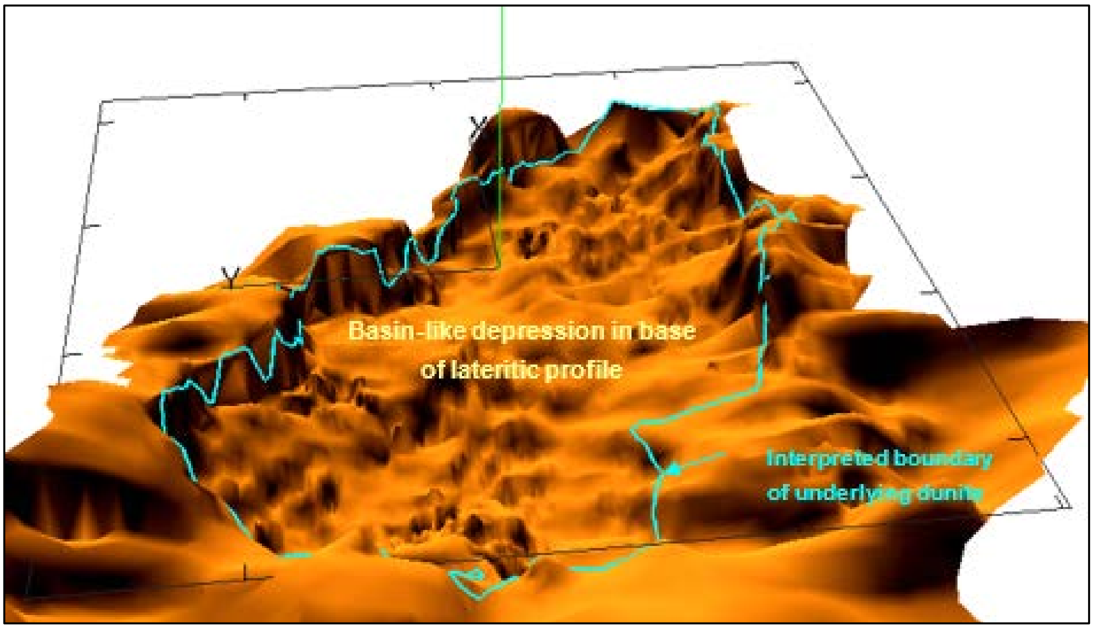



**Figure 7-3: 3D view of base of lateritic profile**

Note: Vertical exaggeration is 10:1.

The lateritic weathering profile is divided into five zones. From top to bottom, these are termed:

1 Residual Overburden (OVB)
2 Transitional Zone (TZ)
3 Goethite Zone (GZ)
4 Silicified Goethite Zone (SGZ)
5 Saprolite Zone (SAP).

The SAP overlies ultramafic bedrock. The bulk of the Ni-Co mineralisation occurs in the GZ and SGZ. The general nature of the profile is diagrammatically shown in Figure 7-4 and the zones are described in more detail below.

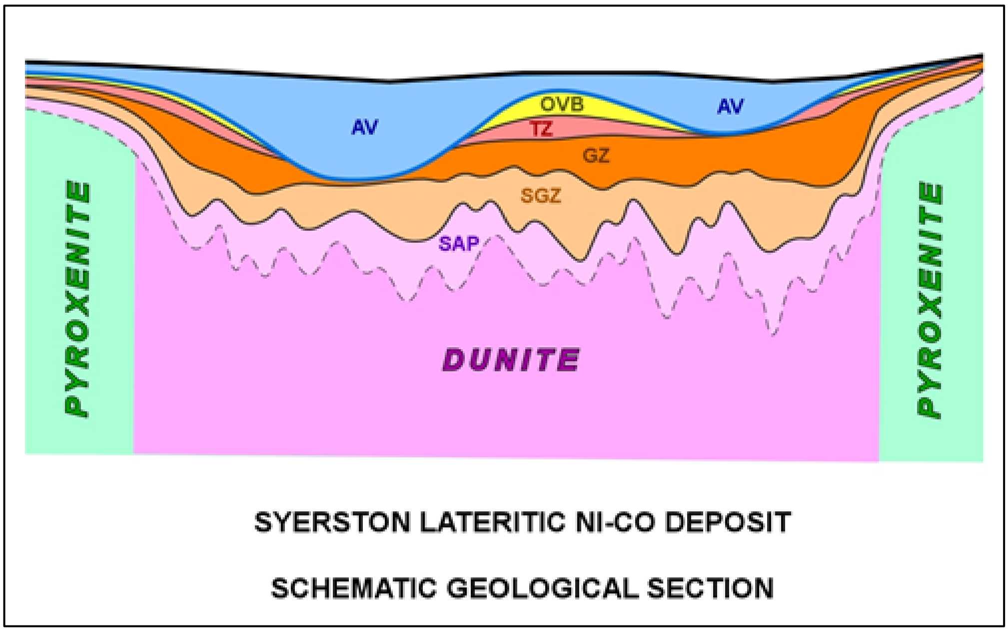



**Figure 7-4: Schematic geological section**

Several distinct Tertiary drainage channels cut across the deposit. These are up to about 40 m deep, but more typically about 20 m - 25 m. These palaeochannels are now filled with alluvium.

The lateritic profile is best developed on the old hilltops between the palaeochannels. This is similar to typical patterns of laterite development in present-day tropical environments, but there may also have been some stripping away of laterite by erosion along the channels when they were active drainage systems.

The main palaeochannel trends roughly grid east, but substantial tributary channels trend northwest, north–south or northeast. The GZ tends to be more or less absent under the main channels, so that they effectively divide the deposit into several parts (Figure 7-5). The main division is along the roughly east–west line of the major palaeochannel, which separates the southeast and northwest parts of the deposit.

The SGZ tends to persist across the whole area, but it is generally thicker below the palaeo-highs.

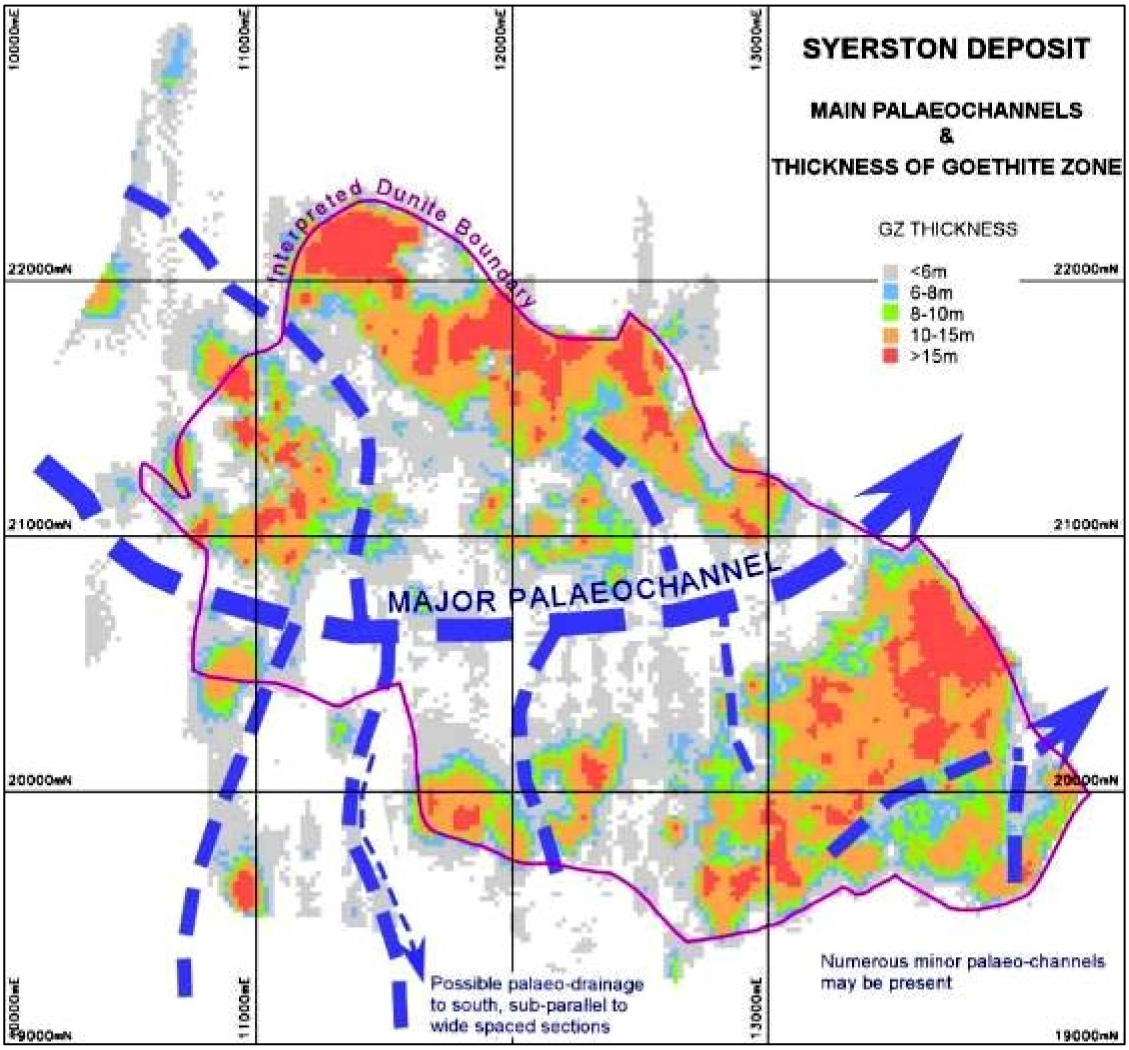



**Figure 7-5: Main palaeochannels and goethite zone thickness**

### 7.2.1 Alluvial cover

The alluvial cover consists mainly of silty clays, with local ferruginous pisolite lenses and gravel lags. The alluvial deposits generally do not contain significant Ni, Co or Pt, although similar deposits further south near the township of Fifield host some Pt concentrations. Near the surface, usually in the top 5 m or so, there is commonly a sub-horizontal layer of magnesite enrichment. This is best developed in the southeastern parts of the area where magnesite mining has occurred in the past.

### 7.2.2 Laterite profile

The following main subdivisions have been recognised in the lateritic profile where it is developed over the dunite core. The same general layering seems to extend laterally for a limited distance over the surrounding pyroxenites, but the whole profile thins markedly and individual units are less clearly defined.

#### Transitional Zone

The uppermost part of the residual profile typically consists of a mixed zone that is usually red-brown and haematitic, and commonly pisolitic near the top, immediately beneath the alluvial cover. It grades down into orange-brown goethitic material. The upper part of the TZ is usually barren or very low grade; from an economic perspective, it will be treated as OVB. The thickness in this zone is quite variable, up to approximately 15 m.

### Goethite Zone

This is a relatively uniform, very fine-grained, orange-brown, goethite-rich layer. Texturally, it resembles clay, but assays and X-ray diffraction (XRD) analyses have shown that it consists mainly of extremely fine goethite. It typically contains more than 40% Fe and usually only about 5% - 10% Si. Minor proportions of Mn oxides are also generally present in this zone, but absent in the TZ. Its upper boundary appears to be quite gradational, while the lower boundary is more abrupt, but potentially quite irregular. The thickness in this zone is variable, up to approximately 20 m.

### Silicified Goethite Zone

This is distinguished from the overlying GZ by a relatively abrupt increase in silica content, with Si values usually in excess of 15% and averaging over 20%. The silica occurs in secondary forms, as laminated veins or sub-horizontal bands and as irregular, coarse meshworks formed where silica has precipitated on joints and fractures. The material between the silica bands is predominantly fine goethite, similar to the overlying GZ. McDonald Speijers understands that field experience has shown that the silica meshworks can sometimes be coarsely vughy towards the base of the zone, which can make drilling difficult. The SGZ is more consistently developed (or preserved) than the GZ, with a typical thickness of 10 - 30 m.

### Saprolite Zone

The saprolite zone represents the weathered top of the underlying bedrock. It is usually a paler brown to grey colour and consists mainly of silica and smectite clays, but relict igneous textures are usually preserved. It only rarely contains significant Ni or Co values, but some elevated Pt grades occur on a very localised basis. The thickness of this zone is unclear because most drill holes stop after penetrating for a few metres.

### 7.2.3 Laterite peripheral to the Dunite core

The 2000 resource model separated a discrete, hard-bounded "peripheral" laterite zone overlying pyroxenites that fringe the dunite core. While there is usually a marked thinning of the lateritic profile across the dunite contact and while grades do tend to drop away quickly from the dunite, McDonald Speijers formed the impression that the weathering profile is not fundamentally different in style over the pyroxenites. While grades decline quite rapidly, this is gradational rather than a sharp step-change. McDonald Speijers therefore interpreted the zones described above as simply extending across the dunite boundary, thinning as the zones moved into the pyroxenite.

### 7.2.4 Bedrock

Fresh basement rocks are rarely intersected in the holes, which generally terminate in the SAP. However, it is often possible to make a reasonable assumption about the nature of the underlying bedrock from textures preserved in the SAP. Interpretations indicate that the dunite core has a discontinuous fringe of olivine pyroxenite, suggesting that the contact between the dunite and the pyroxenites is likely to be gradational to some degree.

### 7.2.5 2016 Mineral Resource estimate interpretation

The criteria used to define the zone boundaries for the current resource are detailed in Table 7-1.

**Table 7-1: 2016 interpretation**

| Zone | Current Interpretations | | |
|---|---|---|---|
| | Zone Code | | Geochemical Boundary Criteria |
| Alluvium | AV | 11 | A marked drop in Si (typically >15% to <10%) and a distinct increase in Cr values |
| Residual Overburden | OVB | 12 | Material below base of AV, but Ni grade below about 0.2% |
| Transitional Zone | TZ | 1 | Upper boundary defined by approximately 2% Ni |
| Goethite Zone | GZ | 2 | Upper boundary defined by Mn greater than 0.35% with Fe greater than 33%, preferably greater than 43% |
| Silicified Goethite Zone | SGZ | 3 | Upper boundary defined by 15% Si |
| Saprolite Zone | SAP | 4 | Upper boundary defined by 6% Mg |
| Peripheral Laterite Zone | | | Not used; where developed, general laterite zones were extended over peripheral pyroxenites |

In the upper parts of the laterite profile, Nickel and cobalt seem to occur as oxide species intimately incorporated into goethite or hematite, possibly at the lattice-structure level. Consequently, the highest Ni grades are in the GZ (which has the highest Fe content), with somewhat lower grades in the lower part of the TZ and in the SGZ. Some subordinate silicate nickel may be present in places associated with smectite, particularly towards the base of the SGZ. Low levels of nickel present in the saprolite would be predominantly in silicates. As is typical in these environments, unusually high cobalt values tend to be closely associated with magnesium oxides which preferentially scavenge this metal.

### 7.2.6 2017 Mineral Resource estimate revised interpretation

A revised geological interpretation of the laterite zones recognisd by McDonald-Speijers (MDS) was carried out. This interpretation used an expanded Chemical Signature in the form of a matrix of both the major and minor element chemistry together with economic element cut-over values. Subsequent global statistics of the laterite zones confirmed the consistency of the approach and the values used.

In addition to the specific chemical criteria outlined in Table 7-2, graphic columns for Fe, Si and Al were placed beside the hole trace in Figure 7-6 to assist in the interpretation by providing a visual comparison of the Fe, Si and Al cut-over values between the various zones in the laterite profile.

In Figure 7-7, the graphics (‘hatch’) columns for Fe, Si and Al were depicted and distinct colour differences were very apparent between the zones. Within the GZ Zone (dark orange hatch), the Fe content (>40% Fe: magenta) directly contrasts with the Si content (<10%; yellow).

The laterite zones have distinctly different chemical signatures as depicted in Figure 7-6 and discussed in the following text.

**AV – Alluvium Zone**

This zone remained as interpreted by MDS and is characterised by high Fe and Si and very high Al. The Ni and Co values are extremely low with mean values of approximately 0.07% and 0.009% respectively.

**OVB – Overburden Zone**

The MDS OVB Zone was not revised. This zone is characterised by Ni values <0.2% Ni and very low Co values (<0.02% Co) with Si values similar or slightly higher than the underlying TZ but relatively higher Al content. The OVB Zone contains mean values for Ni and Co of 0.11% and 0.015% respectively.

**Table 7-2: Chemical signatures utilised – 2017 Resource estimate geological interpretation**

| Transition Zone - TZ | | | Major Elements | Resource Elements | |
|---|---|---|---|---|---|
| | | | | | |
| Fe | | Avg 40% | Red | Ni Avg 0.46% | Red |
| Si | | Avg 7% | Orange | Co Avg 0.03% | Grey |
| Al | | Avg 5% | Magenta | | |
| Mn | | Avg 0.2% | | | |
| Cr | | Avg 4,800ppm | | | |
| | | | | | |
| **Goethite Zone - GZ** | | | | | |
| | | | | | |
| Fe | | >40% | Magenta | Ni Avg 0.75% | Magenta |
| Si | | 5-10% | Orange | Co Avg 0.17 | Magenta |
| Al | | Avg 4% | Red | | |
| Mn | | Avg 1.2% | | | |
| Cr | | Avg 4,100ppm | | | |
| | | | | | |
| **Silicified Goethite Zone - SGZ** | | | | | |
| | | | | | |
| Fe | | Avg 24% | Red | Ni Avg 0.58% | Red |
| Si | | >20% | Magenta | Co Avg 0.07% | Red |
| Al | | Avg 1.3% | Orange | | |
| Mn | | Avg 0.5% | | | |
| Cr | | Avg 4,400ppm | | | |
| | | | | | |
| **Saprolite Zone - SAP** | | | | | |
| | | | | | |
| Fe | | <10% | Grey | Ni Avg 0.28% | Orange |
| Si | | Avg 25% | Magenta | Co Avg 0.02% | Gray |
| Al | | <1.0% | Gray | | |
| Mn | | Avg 0.17% | | | |
| Cr | | Avg 3,000ppm | | | |
| | | | | | |

Note: The Chemical Signature matrix was used as a guide only for the geological interpretation.

### TZ – Transition Zone

The TZ represents weathered GZ material and was defined by the Al values as they increase significantly within the TZ from 2-3% Al to >4%. The Ni values dropped below 0.46% Ni and Co values fell below 0.03% Co compared with the Ni and Co values of 0.75% and 0.17% respectively from the underlying GZ. The mean values of the TZ for Ni and Co are 0.36% and 0.04% respectively.

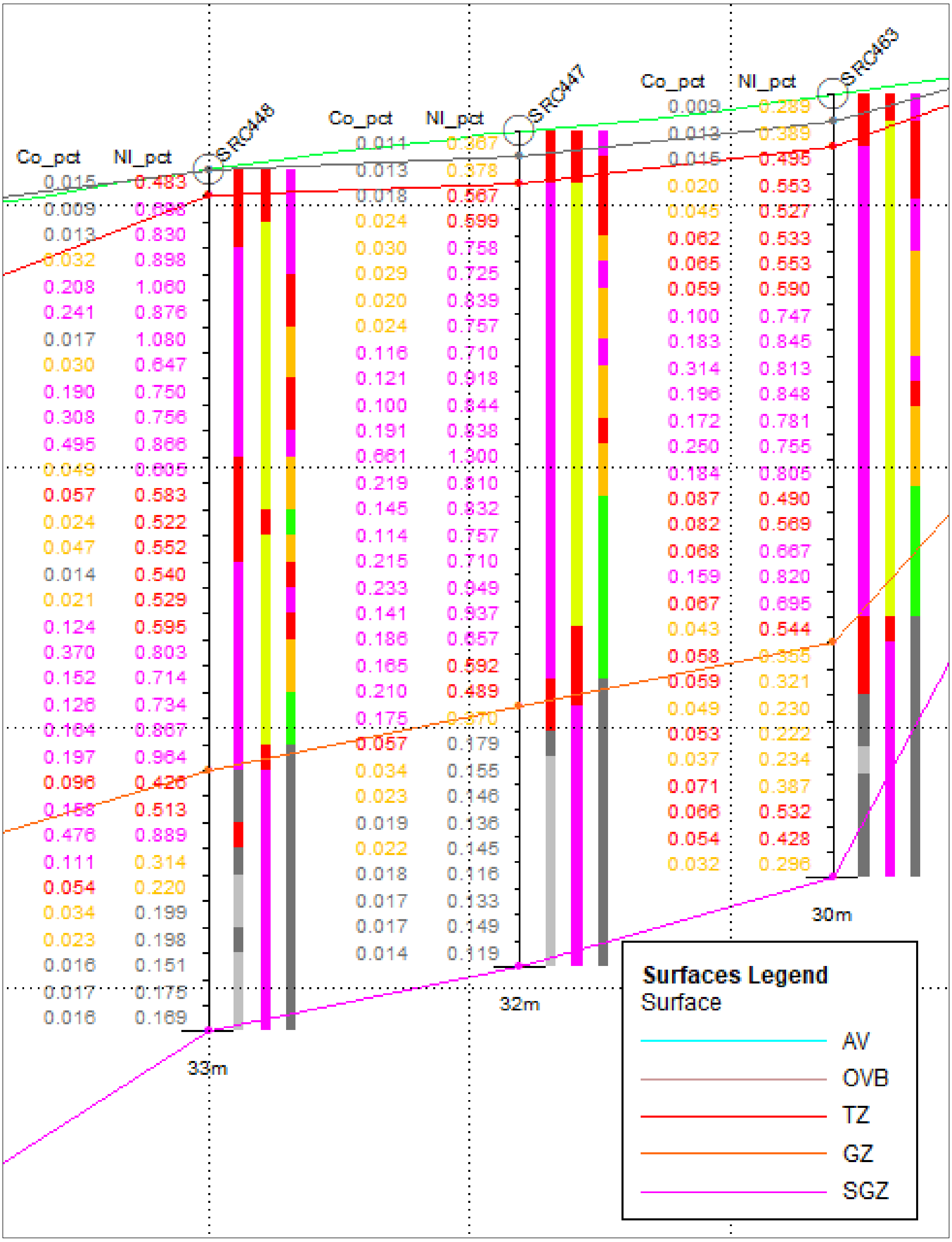
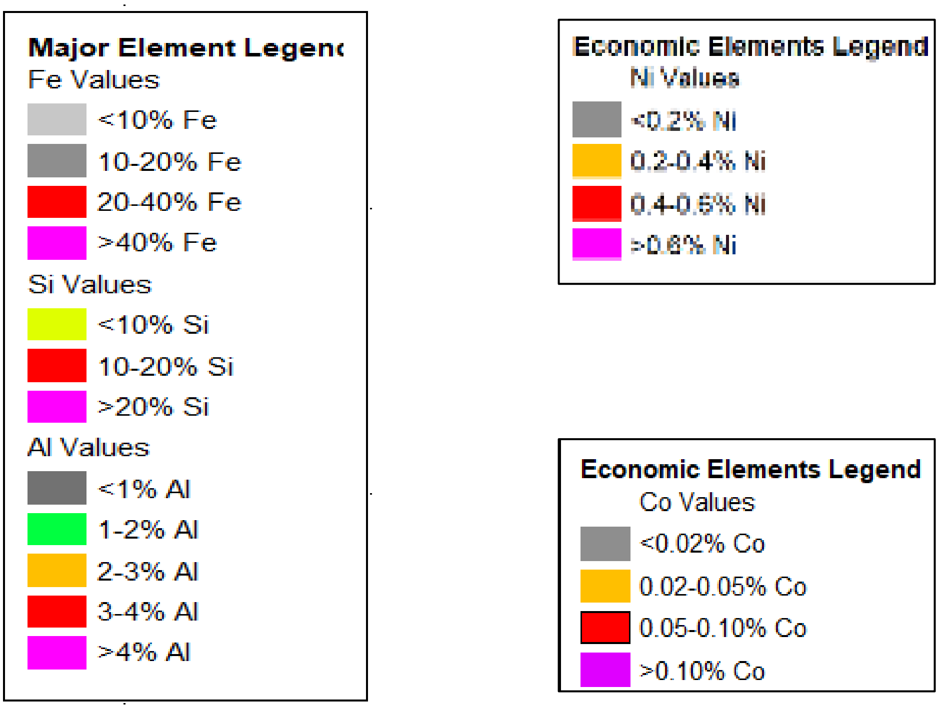



**Figure 7-6: Portion of Section 10,500E with LATZONE**




**Major and Economic Element Values Legend.**

**Figure 7-7: Fe, Si and Al Hatch Columns**

These criteria resulted in the TZ/GZ boundary being lifted relative to that determined by MDS who used the Mn values >0.35% Mn and Fe values >33% as the principal criteria for determining the TZ/GZ boundary.

**GZ – Goethite Zone**

The GZ is characterised by high Fe and low Si and variable Al values. The most significant difference is the increased Ni and Co values where the mean Ni and Co values are 0.75% and 0.15% respectively.

The GZ/TZ boundary is gradational but an Al cut-over value of 2-3% has been used with the result that the mean Al value in the GZ is 3%. The GZ/SGZ is well defined with Si values increasing from approximately 10% Si to >20% Si being the principal criterion.

**SGZ – Silicified Goethite Zone**

The SGZ is characterised by high Si, generally >20% Si and low Al values (<2%). The Ni and Co values are lower than the GZ with the mean Ni and Co values being 0.6% and 0.07% respectively.

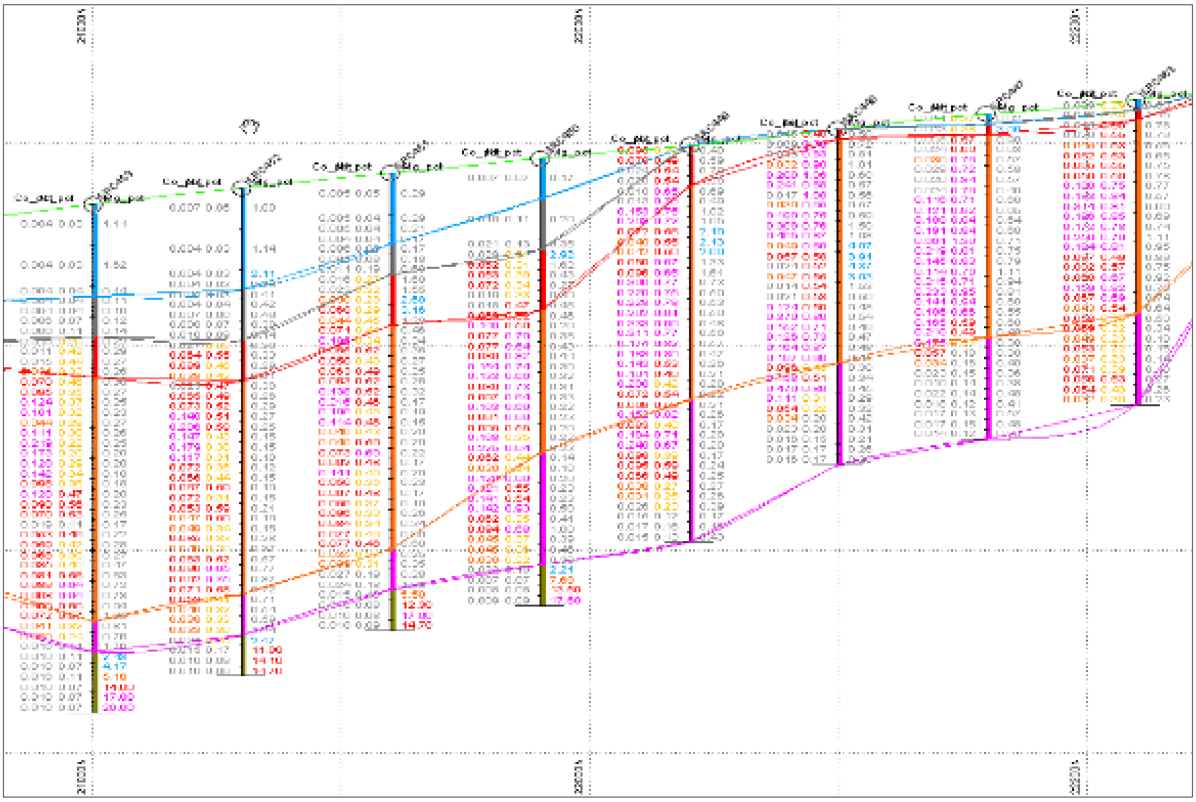


**Figure 7-8: Section 10,500E with geological interpretation and implicit modelling surfaces**

**SAP Zone – Saprolitic Zone**

The SAP Zone represents the saprolite horizon of the underlying dunite source rock. Its principal characteristic is the significant increase in Mg (>5%, see Figure 7.7) together with a commensurate lower Fe content (<10%).

The Ni and Co are lower than the overlying SGZ with the mean Ni and Co values being 0.25% and 0.025% respectively.

**Comments**

MDS highlighted the occurrences of sharp localized rises in the SGZ Zone boundary that may relate to structural features and this phenomenon resulted in an irregular boundary between the GZ and SGZ that would not be practicable during mining.

The revised interpretation recorded these localised rises in the SGZ contact, but for providing a mining as well as a geological boundary, these irregularities were smoothed. During the resource estimation process the grades in this area would reflect this change and would be redirected during scheduling to the appropriate material type.

To further remove any irregularities in the zone boundaries, the snapped points on the drill holes and the wireframes or DTMs created from the interpretation strings were used in an Implicit Modelling (IM) process. Figure 7-8 depicts the outcome of the IM procedure, which honors the data extremely well. The IM surfaces were used to constrain the resource block model.

# 8 Deposit Types

Syerston is a nickel laterite deposit with higher than normal levels of associated Co and local elevated Pt values. Potential ore grades are related to strong concentrations of iron oxides in the laterite profile, which has preferentially developed over the dunite core of a mafic-ultramafic intrusive complex. The deposit falls into the oxide class of laterite deposits rather than the silicate class in which nickel is related mainly to the weathering products of serpentinised ultramafic rocks, such as the hydrated Mg-Ni silicate Garnierite or clay minerals like smectite. The clay content of the main mineralised zones at Syerston is very low.

Although the Fifield district is known for alluvial Pt in deep leads (presumably equivalent to the Tertiary palaeochannels that run across the deposit area), indications of Pt accumulations at or near the base of the alluvial channels that traverse the Syerston deposit are rare; there seems to be little potential for this style of mineralisation in the Project area.

There are indications of some localised potential for bedrock Pt mineralisation in the underlying dunite. However, there is no evidence of any Ni or Co sulphide mineralisation being present in the fresh rocks. Nickel and cobalt mineralisation is intimately associated with Fe oxides, and in the case of elevated Co values also with minor Mn oxides, mainly in the following zones of the laterite profile:

- Lower part of the TZ
- GZ
- SGZ.

These were described in Section 7.2.2; a section showing Nickel and cobalt distribution in relation to the zones is presented in Figure 7-4.

The highest Nickel and cobalt cobalt grades occur in the GZ. Grades decline upwards from the base of the TZ, even though Fe oxide content remains high. This is related to a change in Fe oxide mineralogy from goethite to haematite. Grades are lower in the SGZ largely because of the substantial increase in silica content at the expense of the fine goethite that carries most of the Ni.

At moderate grade levels, Co generally correlates well with Ni and exhibits good lateral continuity. However, the latest round of infill drilling has confirmed that unusually high Co grades (above about 0.4% - 0.5% Co) tend to be localised, with poor lateral continuity (Figure 8-1), particularly in the northwestern part of the deposit. These high values are usually related to concentrations of Mn, which may in turn be related to relicts of steep structures where Mn was initially deposited in and around bedrock features, such as joints or shears, during the early stages of weathering.

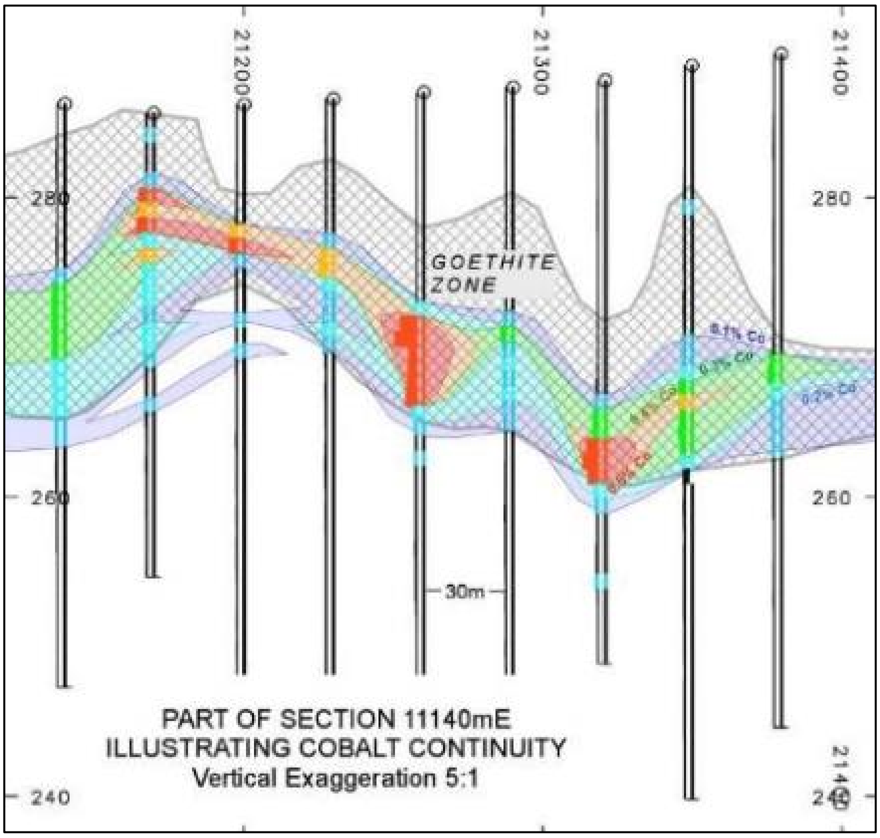



**Figure 8-1: Section illustrating continuity of cobalt mineralisation**

There appear to be differences in the spatial distribution patterns of high Co values in different parts of the deposit. In the southwest part, where extremely high values are less frequent, the Co tends to form relatively extensive, sheet-like layers within the GZ. In the northwestern part, high Co values are more restricted in extent (but with extremely high values being more common) and they tend to occur as fringes around pronounced highs in the top of the SGZ (Figure 8-2). There are also indications in the northwestern part of the deposit that some of very high Co concentrations might be localised in depressions in the upper part of the SGZ.

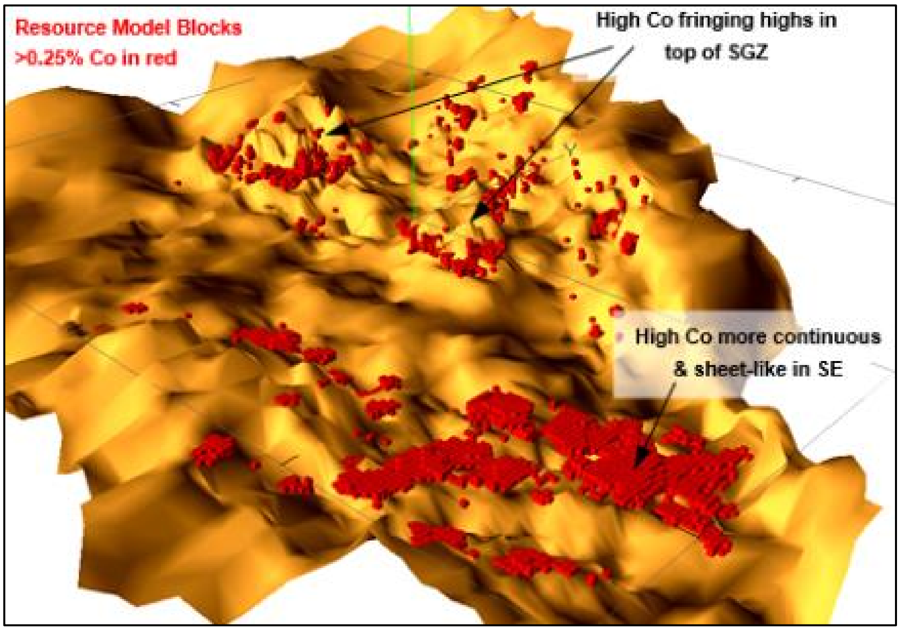



**Figure 8-2: Cobalt distribution relative to top of SGZ**

The localised nature of unusually high cobalt grades means that relatively wide-spaced pre-production drilling patterns can result in quite large volumes of consistently high cobalt values being predicted around individual drill holes in resource models, when in reality, the high cobalt occurs in numerous much smaller patches distributed over a larger area. This distribution pattern will need to be considered during open pit optimisation and mine planning to avoid generating small starter pits that are heavily dependent on a single drill hole.

In a broader sense, the lateral distribution of nickel and cobalt is heavily influenced by both the alluvium-filled Tertiary palaeochannels that cut across the deposit (Figure 7-5) and the extent of the underlying dunite core to the intrusive complex.

Figure 8-3 and Figure 8-4 show a plan view of the distribution of Nickel and cobalt in the two main mineralised layers (GZ & SGZ).

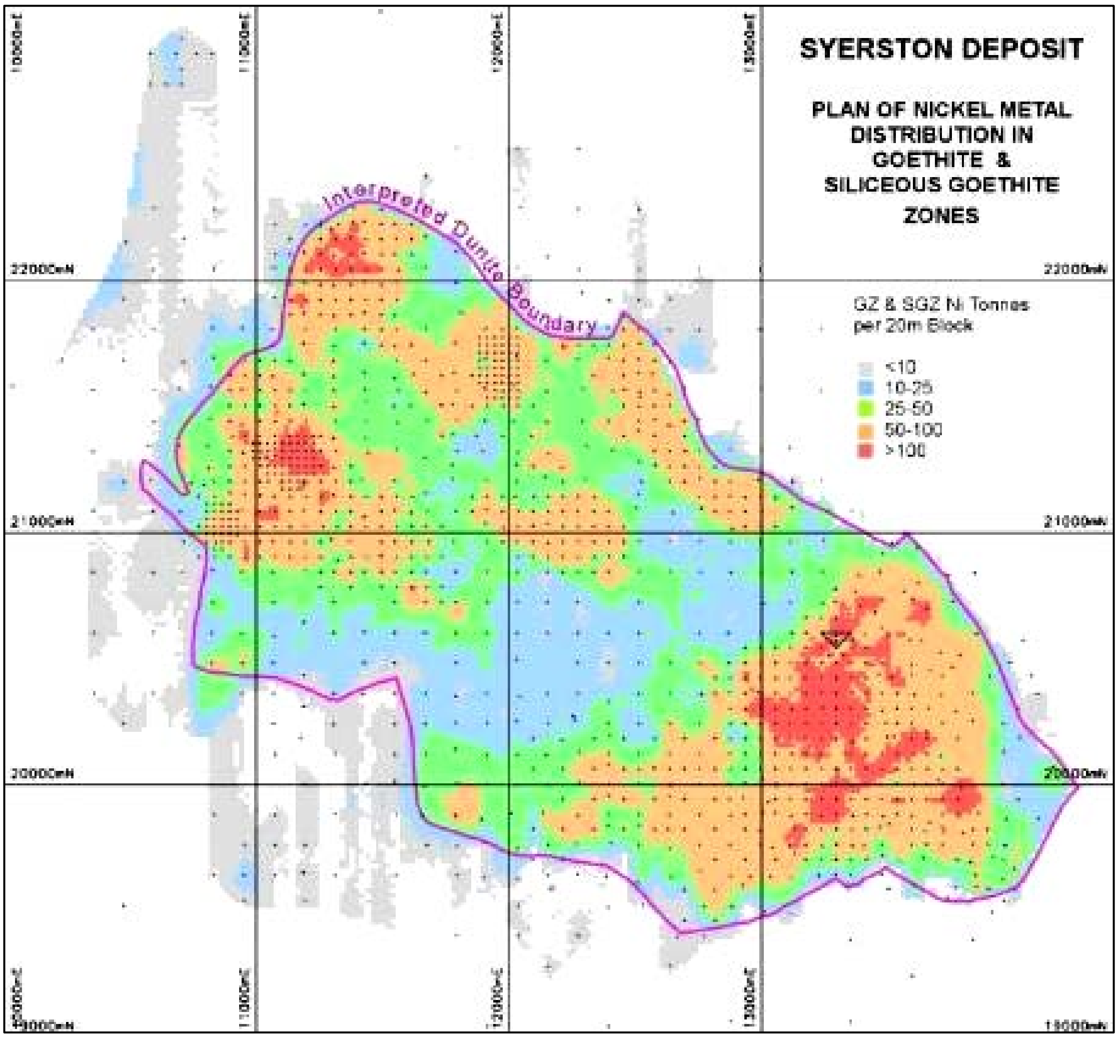



**Figure 8-3: Nickel metal distribution in GZ and SGZ**

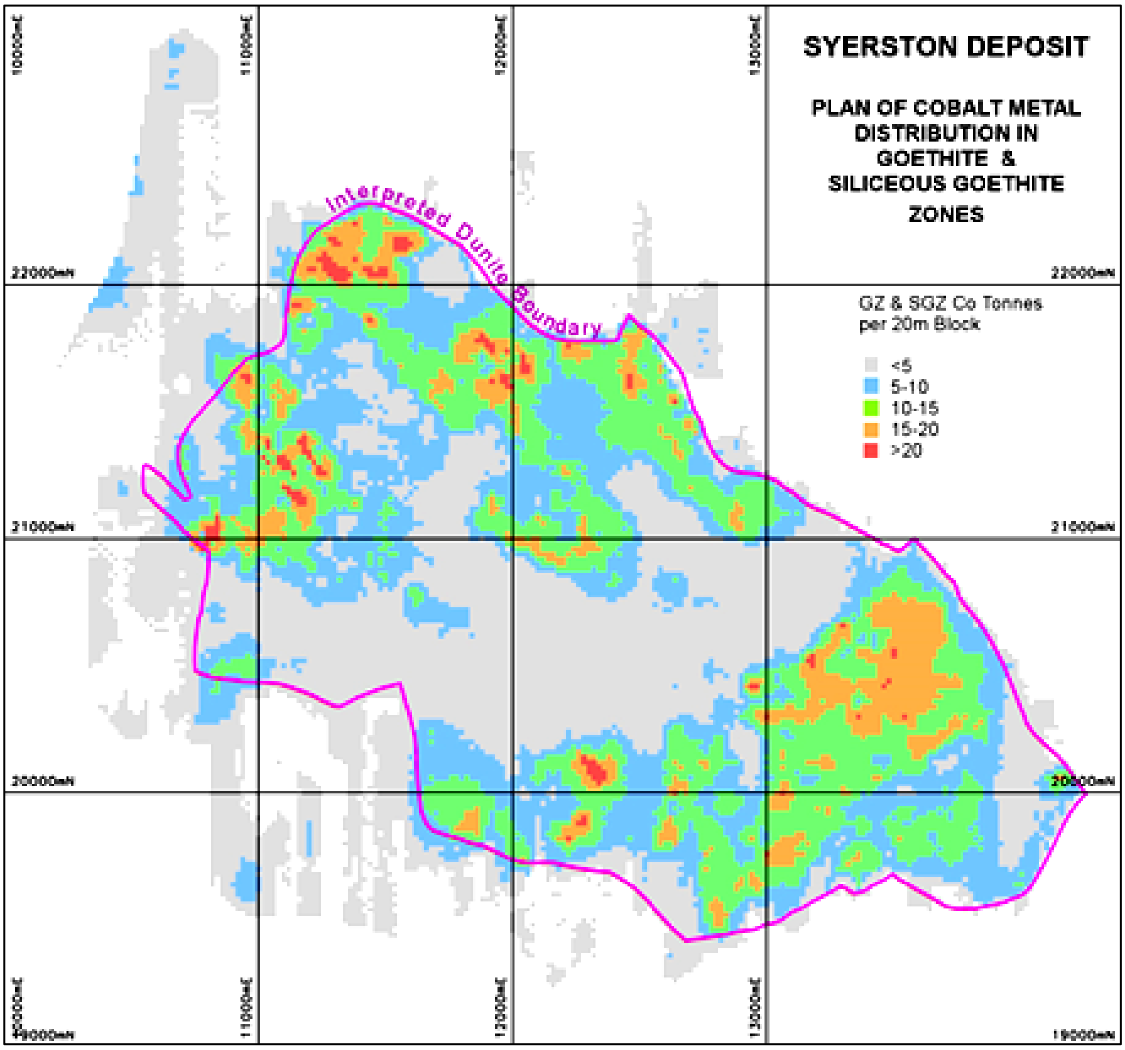



**Figure 8-4: Cobalt metal distribution in GZ and SGZ**

# 9 Exploration

Exploration contributing to the database currently available for resource estimation commenced in about 1986, initially targeting Pt. The potential for lateritic Ni-Co was apparently recognised in about 1988 - 1989, but most of the work programs aimed at delineating this mineralisation were carried out after 1995.

The principal exploration and delineation tool has been drilling; for the most part, this has involved relatively shallow, vertical RC percussion holes. No recent programs of other types of exploration have been carried out.

# 10 Drilling

The numerous phases of drilling that have occurred at Syerston are summarised in Table 10-1.

A total of 1,733 holes totalling 61,736 m were available to SRK as at 1 February 2016. Of those, 1,363 holes aggregating 49,883 m were judged acceptable for use in resource estimation. These are overwhelmingly RC holes, with a small proportion of aircore holes (about 3% of the total) accepted only where there were substantial gaps in the RC drilling pattern.

The location of the drill holes is shown in Figure 10-1 and Figure 10-2.

## 10.1 Drilling methods

Prior to 1994, open hole RAB drilling was used. This method is highly prone to sample contamination, and collar coordinates for these holes are also of doubtful accuracy. Some holes were taken into account during the interpretation of laterite zones, but McDonald Speijers considered them to be unacceptable for use in resource estimation and, at that stage, they were excluded as a whole.

RC aircore drilling was introduced in 1994. The use of this method reduces the potential for sample contamination, but the rigs used were commonly unable to penetrate the SGZ, rendering many of the holes of limited value. They were considered during laterite zone interpretation, but only a small proportion of the holes were accepted for use in resource estimation in areas where no subsequent RC drilling had been carried out.

In late 1997, larger RC rigs were introduced using face-sampling downhole hammer bits. These were generally capable of fully penetrating the laterite profile. The overwhelming majority of holes that were accepted for use in resource estimation were of this type.

McDonald Speijers was advised that similar rigs were used for the 2014 - 2015 RC drilling.

Very little diamond core drilling has been done – 13 shallow, vertical, diamond core holes were drilled, principally to obtain samples for density measurement and preliminary metallurgical testwork. Average core recovery was about 85%. The diamond holes were also excluded because they did not penetrate the complete laterite profile.

Nine large diameter (770 mm) Calweld holes were drilled in 1999 to obtain bulk samples for metallurgical testwork and bulk density measurements.

**Table 10-1: Summary of drilling programs**

| Time period | Operator | Hole type | Total | | | Used in 2016 Resource Model | | | Comments |
|---|---|---|---|---|---|---|---|---|---|
| | | | Hole number range | Number of holes | Metres in database | Hole number range | Number of holes | Metres in database | |
| Sep 1986 - Mar 1988 | Noble Resources | RAB | Not present | 0 | 0 | | | | Not in database |
| Mar 1988 - Mar 1989 | Noble-Poseidon JV | RAB | CMD1-5 | 5 | 237 | CMD1-5 | 0 | 0 | Early Pt exploration? |
| | | RAB | FR1-105 | 105 | 3,702 | FR1-105 | 0 | 0 | Pt exploration; laterite potential recognised? Holes renamed as "FR" series; reports state these were resampled and renamed as the "FRRS" series, but the database shows different coordinates for FR & FRRS holes with the same number |
| | | RAB | FRRS1-82 | 82 | 2,440 | FRRS1-82 | 0 | 0 | All shown with the same start and end dates (23/09/88 & 22/03/99) and lacking details of the drilling contractor. However, co-ordinates are not the same as matching FR series numbers. |
| | | RAB | P30 & 36 | 2 | 73 | P30 & 36 | 0 | 0 | Start and end dates and drilling contractor as above. No assay data. Same depths and within 20 m in co-ordinates of FR30 & 36. Suspected to be phantom holes remaining from renaming of the "P" series. |
| Sep 1989 - Mar 1990 | Noble-Poseidon JV | RAB | Not present | | | Not present | | | Magnesite exploration? |
| | | RAB | FR106-115 | 10 | 334 | FR106-115 | 0 | 0 | Metallurgical samples, mineralogy, ICP scans |
| Sep 1991 - Sep 1992 | Noble-Poseidon JV | DDH | FPD1-2 | 2 | 332 | FPD1-2 | 0 | 0 | Bedrock Pt targets |
| Aug 1993 - Aug 1994 | Noble Resources | RAB | SRB117-119 | 3 | 105 | SRB117-119 | 0 | 0 | |
| | | Aircore | SAC120-139 | 20 | 603 | SAC120-139 | 11 | 316 | SAC120-124 suspect; most holes replaced by later RC holes |
| Aug 1995 - Aug 1996 | Uranium Australia | Aircore | SAC140-267 | 128 | 3,672 | SAC140-267 | 34 | 1,125 | Infill in laterite area; most holes replaced by later RC holes |
| Aug 1997 - Aug 1998 | | RC | SRC001-340 | 341 | 14,149 | SRC001-340 | 339 | 14,069 | Infill drilling |

| Time period | Operator | Hole type | Total | | | Used in 2016 Resource Model | | | Comments |
|---|---|---|---|---|---|---|---|---|---|
| | | | Hole number range | Number of holes | Metres in database | Hole number range | Number of holes | Metres in database | |
| | Uranium Australia | DDH | SDD1-5 | 5 | 169.1 | SDD1-5 | 0 | 0.0 | PQ3 core for density and metallurgy tests' all holes sampled and assayed |
| Aug 1998 - Oct 2000 | Black Range Minerals | RC | SRC341-1076 | 732 | 25,857 | SRC341-1076 | 724 | 25,697 | Infill drilling; additional holes drilled after Aug 1999 are not reported in FS documents |
| | | DDH | SDD6-13 | 8 | 319.1 | SDD6-13 | 0 | 0.0 | PQ3 core for density and metallurgy tests; holes SDD6,8,9,10 & 12 were not assayed |
| | | Calweld | SCW1-9 | 9 | 234 | SCW1-9 | 0 | 0 | Large diameter holes; three holes in Nov 1999 were not reported in the FS documents |
| Feb 2005 - Mar 2005 | Ivanplats Syerston | RC | SRC1077-1193 | 117 | 4,321 | SRC1077-1193 | 117 | 4,321 | Local 30 x 30 m infill drilling |
| | | | SRC1194-1225 | 32 | 1,594 | SRC1194-1225 | 32 | 1,594 | Infill drilling around elevated Pt intercepts |
| | | | SRC1226-1251 | 26 | 833 | | 0 | 0 | Twinning previous Calweld & RC holes |
| Aug 2014 | Ivanplats Syerston | RC | SRC1263-1276 | 14 | 381 | SRC1263-1276 | 14 | 381 | Testing for peripheral Sc zones |
| Apr 2015 | Scandium21 | RC | SRC1277-1310 | 34 | 944 | SRC1277-1310 | 34 | 944 | Infill drilling in areas of elevated Sc |
| Nov 2015 | Scandium21 | RC | SRC1311-1368 | 58 | 1436 | SRC1311-1368 | 58 | 1436 | Infill drilling in areas of elevated Sc |
| Project Totals | | RAB | | 207 | 6,891 | | 0 | 0 | |
| | | Aircore | | 148 | 4,275 | | 45 | 1,441 | |
| | | RC | | 1,354 | 49,515 | | 1,318 | 48.442 | |
| | | DDH | | 15 | 820 | | 0 | 0 | |
| | | Calweld | | 9 | 234 | | 0 | 0 | |
| | | ALL | | 1,733 | 61,736 | | 1,363 | 49,883 | |

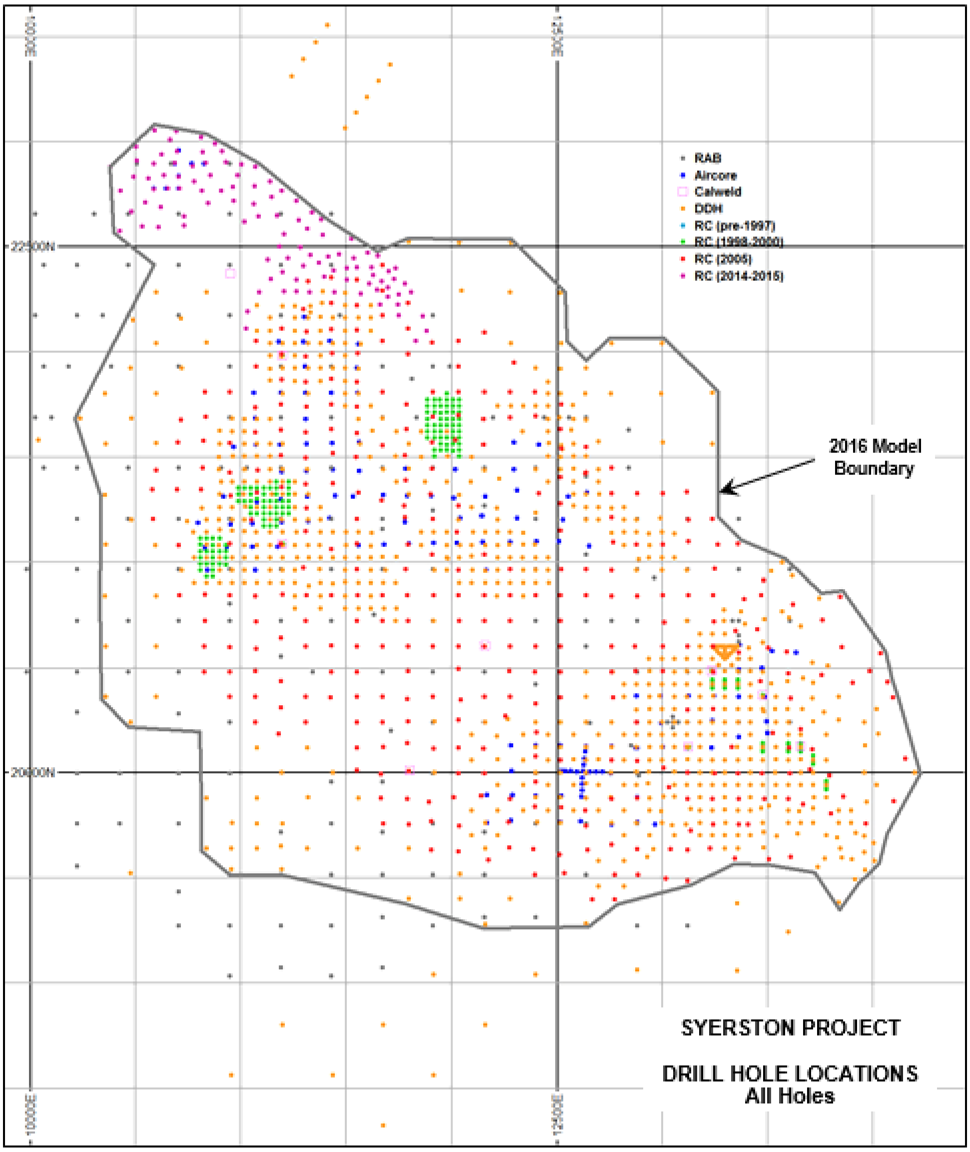



**Figure 10-1: Drill hole location plan - all holes**

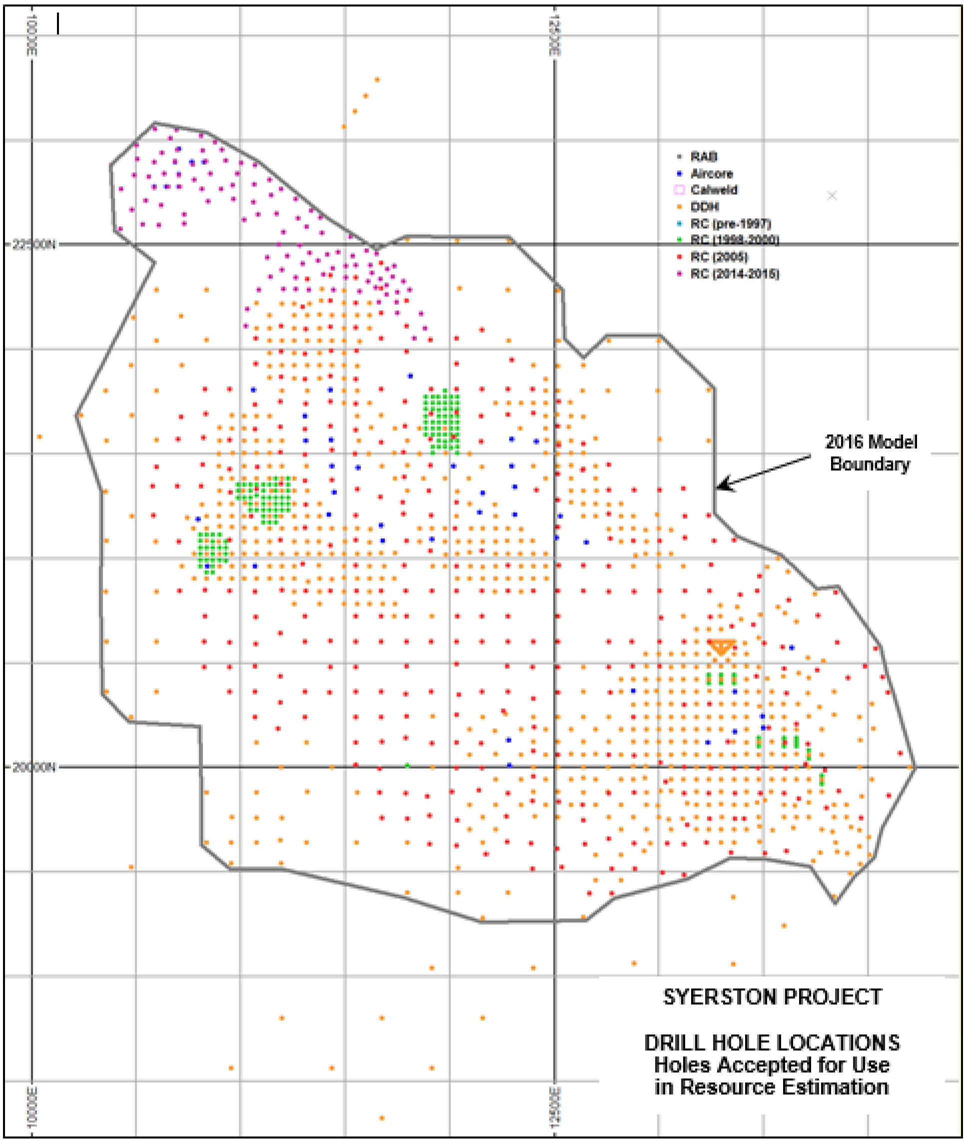



**Figure 10-2: Drill hole location plan - holes used in resource grade estimation**

## 10.2 Drilling patterns and hole orientation

By 2000, a drilling pattern of approximately 120 x 120 m had been completed over much of the resource area, with infill drilling to approximately 60 x 60 m over some of the better mineralised portions making up about one third of the total area.

In early 2005, infill drilling to 30 x 30 m was conducted in the areas of some Year -1 pit designs generated during the 2000 FS. These had focused on areas of unusually high Co grades in the resource model of the time. The closer-spaced drilling was intended to either confirm the continuity of high Co values or to better constrain their volumetric influence (Figure 10-2).

With the exception of two diamond drill holes from 1991 - 1992, all holes have been vertical, with an average depth of ~35 m. This hole orientation was appropriate for the delineation of the lateritic Nickel and cobalt resource. However, at the current hole spacings, vertical holes do not support reliable estimation of average Pt grades.

In 2014 - 2015, RC drilling coverage was extended to the northwest, principally to investigate Sc potential over peripheral pyroxenites. The drilling pattern was somewhat irregular, but approximated 60 x 60 m (Figure 10-2).

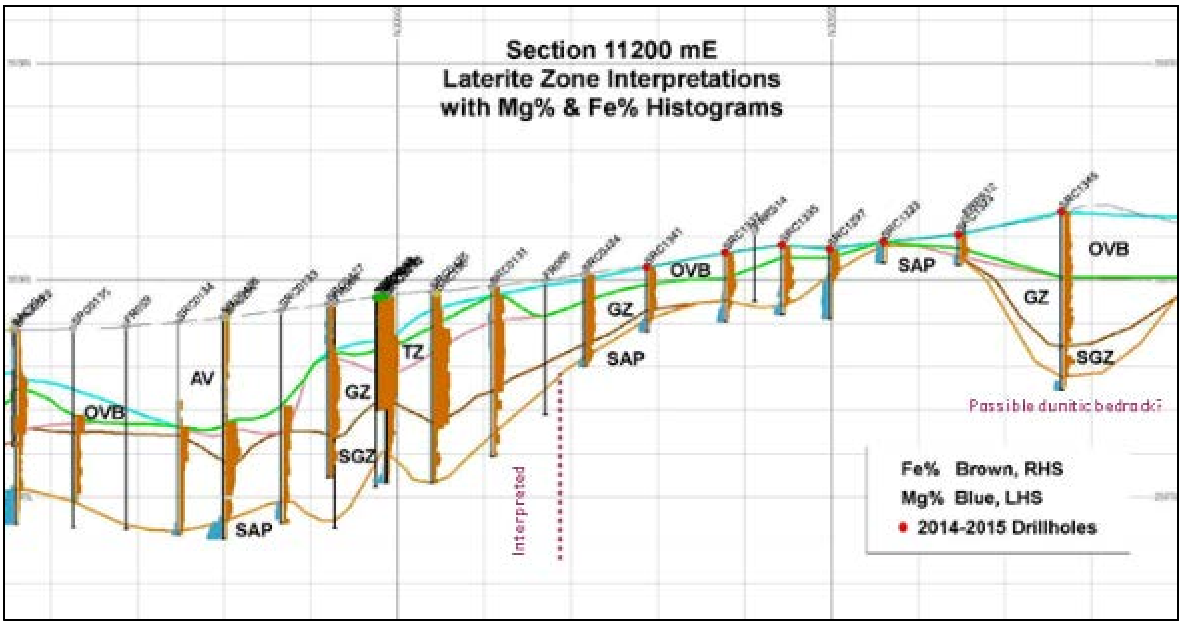



**Figure 10-3: Section 11200 mE - laterite zone interpretations with Mg% and Fe% histograms**

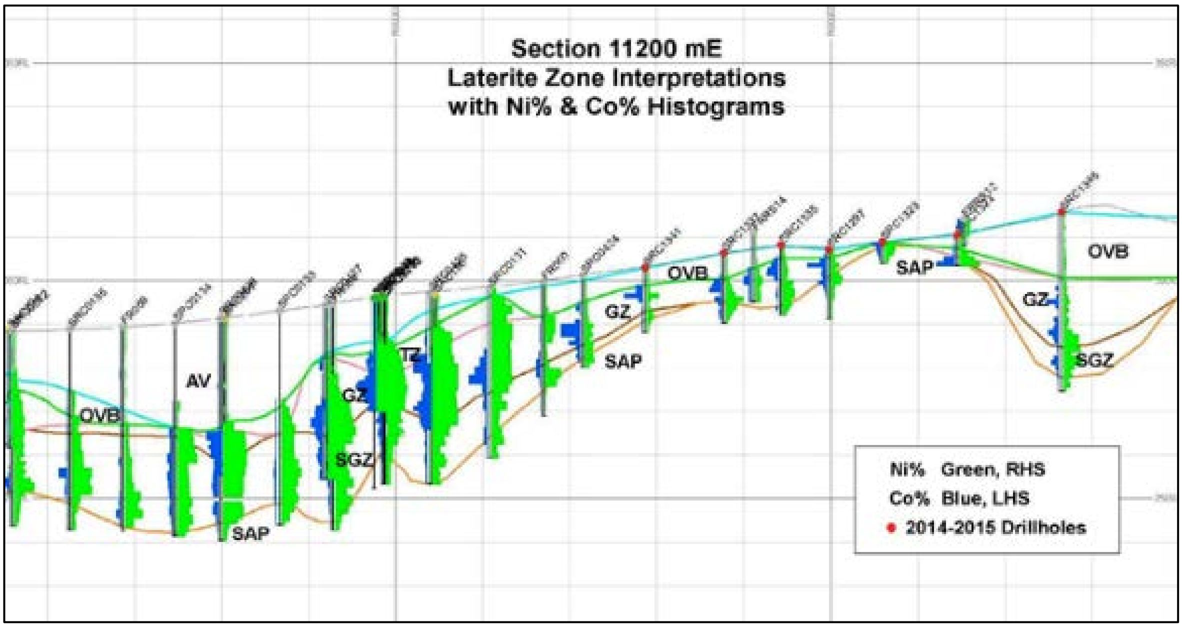



**Figure 10-4: Section 11200 mE - laterite zone interpretations with Ni% and Co% histograms**

## 10.3 Comparison of drilling methods

In 2005, McDonald Speijers initially tried to compare results from different drilling methods using pairs of holes with collars less than 6 m apart. However, in almost all cases, too few pairs were available to provide reliable comparisons and the results were inconclusive.

McDonald Speijers then selected equivalent patterns of aircore and RC holes that covered the same areas and compared their overall sample assay frequency distributions. This involved a much larger number of holes (63 holes of each drill type), but the comparison was limited to the TZ and GZ which had been more or less fully penetrated by both sets of holes.

This indicated that there might be a significant and persistent bias between aircore and RC holes for both Nickel and cobalt. On average, aircore grades tended to be lower by around 10% relative for Ni and 15% relative for Co.

The most reliable measurements of in situ Ni-Co grades came from several large diameter Calweld holes that were bulk sampled. An initial comparison of grades between bulk sampled Calweld holes and the nearest existing RC holes involved separation distances of up to 9 m (averaging over 6 m). Results for nickel were satisfactory, but average cobalt grades in the RC holes were 30% - 40% relative higher. Coupled with the aircore comparison, this raised concerns about the possibility of sampling biases in results from RC holes.

To further investigate this possibility, in early 2005, multiple RC twin holes (four in each case) were drilled around five Calweld holes that had been properly bulk sampled. The new RC holes were all collared within ~1.5 m of the Calweld holes.

A comparison of the average result of four RC twins with bulk sample results from each Calweld hole showed satisfactory comparisons for both Nickel and cobalt. Average grades varied by less than ±5% relative between the Calweld and RC holes.

At the same time, the old RC holes located near the Calweld holes were also twinned with collar separations of ~1.5 m. The results were also satisfactory for both nickel and cobalt, average grades varying by less than ±5% relative between the two sets of RC holes.

**Table 10-2: 2005 RC vs Calweld twinning - summary of comparison**

| | Pairs | Total metres | Grade (Ni%) | Grade (Co%) | Total metres | Grade (Ni%) | Grade (Co%) |
|---|---|---|---|---|---|---|---|
| | | New RC holes (average of four RC twin holes for each Calweld hole) | | | Calweld bulk samples | | |
| All Pairs | 5 | 90 | 0.89 | 0.13 | 88.82 | 0.88 | 0.13 |
| Relative Difference | | 1.3% | 0.9% | 4.4% | | | |
| | | **New RC holes** | | | **Old RC holes** | | |
| All Pairs | 7 | 156 | 0.75 | 0.12 | 156 | 0.74 | 0.12 |
| Relative Difference | | 0% | 1.8% | -4.3% | | | |

The results of the 2005 twinning program indicated that RC holes were probably free of bias for both Nickel and cobalt, and should provide a satisfactory basis for the estimation of Nickel and cobalt resource grades.

McDonald Speijers concluded that previous poor Co grade comparisons between bulk sampled Calweld holes and the nearest existing RC holes at the time were due primarily to inherent, short-range, Co grade variability. This has implications for future grade control requirements should a Ni-Co project proceed.

The possibility remains that a bias exists between RC and aircore holes. If so, indications are that the use of a small proportion of aircore holes would tend to result in a slight underestimation of average grades for both Nickel and cobalt. Given the low number of aircore holes used, any impact is likely to be far within the usual limits of error for resource estimates.

SRK's 2005 report contains further details of comparisons between the drill hole types.

## 10.4 Drill hole sampling

The majority of drilling was completed by early 2000. McDonald Speijers was only able to observe drilling and sampling procedures for infill RC holes drilled in early 2005.

### 10.4.1 Sampling procedures

#### Aircore holes

Aircore holes were sampled over 2 m intervals. Samples were apparently split in the field to ~2 kg, but no record of procedures could be located.

#### RC holes: 1997 - 2000

Sampling procedures for the majority of RC holes were well recorded in comprehensive standard operating procedures documented by Black Range. These applied to holes from SRC341 onwards. No information was available for earlier holes. During this period, RC holes were sampled over 1 m intervals. Total sample was collected via rig-mounted cyclones and riffle split to about 2 - 4 kg, using 3-tier multi-stage riffles. Splits were placed in plastic bags, and marked with the hole number and depth. Total reject was bagged in plastic, labelled and weighed. As described, these procedures should have been satisfactory and consistent with normal industry practices.

An apparently very small, but unquantified, proportion of wet intervals were split by spear or grab sampling. These samples would not be reliable.

In 1999, SLA independently observed some RC drilling and sampling, and confirmed that procedures *"followed the protocols detailed as standard operating procedures".*

#### RC holes: 2005

Samples were collected over regular 1 m intervals. All cuttings were collected by a rig-mounted cyclone and, at the end of each sample interval, discharged into a rig-mounted 3-tier riffle. The subsample split was collected in a small plastic bag and the reject in a large plastic bag. Plastic sample bags were labelled, stapled shut and wrapped with tape to secure them for transport, before being placed into poly-weave sacks. Bagged reject was labelled and laid out near the drill hole collar.

McDonald Speijers observed some drilling in the early stages of the 2005 program and was satisfied that field sampling practices complied with standard procedures specified by Ivanplats and were consistent with normal industry practices. The procedures were similar to those reportedly used previously by Black Range and should have produced samples of similar quality. Samples generally seemed to be of good quality in the mineralised zones.

#### RC holes: 2014 - 2015

According to OreWin, cuttings from RC holes drilled in 2014 were collected through a cyclone over 1 m intervals. A sample was split from each 1 m interval using a riffle mounted on the rig under the cyclone and initially these were composited over 2 m intervals. Subsequently, 1 m samples were collected. Bulk rejects were stored near the collars in large, labelled plastic bags.

As described, this procedure was similar to those used during previous RC drilling programs conducted since 1997, except for the initial use of 2 m sample intervals.

Procedures in 2015 were supposed to have been similar to those followed in 1998 - 2000 and in 2005, but McDonald Speijers was unable to independently verify this because the contract geologist who supervised the programs was out of the country and could not be contacted.

#### Diamond drill holes

The proportion of diamond drill holes is far too small to have any significant influence and the diamond holes were excluded from use in resource grade estimation. Handling of the core in the field was well documented by Black Range and appeared to have been satisfactory. Total core was transported to the laboratory before nominated intervals were sampled. This was apparently done by crushing and rotary splitting, but no documentation of this procedure could be located.

### Calweld holes

Field sampling procedures for the large-diameter Calweld holes were well documented, including a good photographic record. The hole diameter was approximately 770 mm. A length of 0.6 - 0.8 m at a time was placed in total into large, heavy-duty bags and a grab sample of ~10 kg was taken and placed in a sealed plastic bag. The large bags were weighed and placed in a storage area adjacent to the holes. Sample recovery seems to have been virtually 100%.

SLA independently observed some of the Calweld drilling and reported that, in its opinion, the Calweld drilling produced very good samples. Sample preparation and sub-sampling procedures for the field grab samples were not documented. Bulk samples from selected mineralised intervals were transported in their entirety to a metallurgical laboratory in Perth, Western Australia, where McDonald Speijers understands they were crushed in total and then rotary split. However, no documentation could be located.

## 10.4.2 Sample recovery

### Aircore holes

No information could be located about sample recoveries from aircore holes.

### RC holes

There were no available records of sample recovery for RC holes drilled before late 1998. In 1998-2000, sample weights were measured in the field for RC holes from SRC341 to SRC1004 using simple bathroom scales. These scales proved to be quite unsuitable and Black Range concluded that the sample weight information was unreliable. Weighing was discontinued from hole SRC1005 onwards.

Weighing RC samples is often good practice, but McDonald Speijers is doubtful about its value in this case. Substantial variations in bulk density, extremely large variations in the expansion characteristics of the materials (typically GZ expands far more when drilled than other materials) and unpredictable voids (particularly in the SGZ) meant that McDonald Speijers was not satisfied that either weight or volume measurement would reliably quantify RC sample recovery. In 2005, after monitoring some of the drilling, McDonald Speijers concluded that a visual estimate of percentage sample recovery made on the spot by the geologist at the drill rig would provide an equally meaningful measure of recovery as weighing.

Both Black Range and SLA geologists who observed RC drilling in the 1998 - 2000 period commented that sample recovery appeared to be satisfactory, except for some intervals in the SGZ. McDonald Speijers' field observations in 2005 were similar, and estimated sample recoveries for the 2005 RC holes confirmed that poor recoveries were uncommon and generally restricted to the SGZ or the underlying SAP (Table 10-3).

In 1999, SLA noted that significant proportions of the RC sample weights recorded in all geological zones in the 1998 - 1999 period were lower than might have theoretically been expected. SLA became concerned about the possibility of grade biases being generated if sample recovery was poor, and in late 1999, SLA engaged Snowden Mining Industry Consultants (Snowden) to conduct an independent review of RC sample recovery.

In McDonald Speijers 's experience, smaller samples than theoretically expected are quite a common phenomenon when drilling poorly consolidated materials.

**Table 10-3: Estimated sample recoveries by zone – 2005 RC drilling**

| Modelled zone | Average estimated recovery |
|---|---|
| AV | 94% |
| OVB | 95% |
| TZ | 94% |
| GZ | 94% |
| SGZ | 87% |
| SAP | 89% |

In its report dated January 2000, Snowden concluded that there was no apparent bias for Co and possibly a modest positive bias for Ni in the GZ and TZ only, but that any potential impact on resource grade estimates should be well within normally acceptable limits of error. However, the sample weight and bulk density data used were all affected by inherent technical limitations and potential errors, and Snowden qualified its conclusions by noting that any apparent biases could simply be artefacts of the data.

Snowden's analysis was also hampered by a pronounced negative correlation that exists between bulk density (and therefore sample weight) and both Nickel and cobalt grades.

At the beginning of 2005, concerns were still held about the possibility of sampling biases in RC holes. Potential for biases was confirmed by sizing analysis of RC cuttings, but satisfactory results obtained from the 2005 program of close-spaced RC twinning around bulk sampled Calweld holes strongly indicated that, on average, RC samples were free of significant sampling biases.

## Diamond drill holes

Core recovery was reportedly erratic. Logs and photographs indicated that it tended to be either near 100% or near zero. On average, recovery was about 85% through the main mineralised zones.

## Calweld holes

Sample recovery from the Calweld holes was reported to be virtually 100%.

# 11 Sample Preparation, Analyses, and Security

Up until 1998, Australian Laboratory Services Pty Ltd (ALS), located in Orange, New South Wales, was the primary laboratory for sample preparation and assaying, although there is mention in some older Annual Reports that the ALS laboratory in Brisbane, Queensland, was also involved to some unknown extent.

From late 1998 onwards, Ultratrace Analytical Laboratories (Canning Vale, Western Australia) became the primary laboratory, with Genalysis Laboratory Services (Genalysis) in Maddington, Western Australia, used for check assays. This remained the case in 2005.

In 2014 – 2015, samples were assayed at ALS in Brisbane, Queensland, after sample preparation at ALS' facilities in Orange, New South Wales.

In 1999, significantly mineralised intervals from previous aircore holes (>0.4% Ni) and from RC holes up to SRC340 drilled between 1995 and 1998, were re-assayed at Ultratrace. The intervals involved were apparently selected on the basis of a Ni equivalent value >0.25%, apparently using the formula NiEq%= Ni% + (Co% x 3.64).

## 11.1 Sample despatch and security

No details could be located about sample despatch procedures prior to 1999.

### 11.1.1 1999 - 2000

In 1999 and 2000, procedures involved the following steps:

- The outside of each sample bag was marked using a combination of hole number and sample depth as a sample number.
- Five samples at a time were placed into larger plastic bags and then into large "bulka" bags.
- Sample submission forms detailing the date, number of bags, number of samples and sample numbers, were completed. One copy was sent to the assay laboratory by fax and another was sealed in a plastic bag and inserted into one of the sample bags (specifically marked with coloured tape).
- The "bulka" bags were collected daily by a local transport contractor and trucked to the laboratory in Perth.

The lack of a durable sample ticket in each sample bag would have increased the potential for some mix-ups to have occurred at the sample preparation stage, but sample numbering and despatch procedures were adequate.

Prior to 2005, no particular security measures were imposed during sample transport between the drill rig and the laboratory. Consequently, the possibility of outside interference cannot be totally excluded for this time period, but the style of the Nickel and cobalt mineralisation is not particularly amenable to tampering. An independent custody sampling check was conducted by SLA in 1999, which indicated that tampering was unlikely to have occurred.

### 11.1.2 2005

In 2005, procedures involved the following steps:

- The same sample numbering system as in 1999 - 2000 was used, with the number marked on the outside of each sample bag.
- The sample bags were folded and stapled shut, then wrapped securely with a piece of tape, before being placed about five at a time into large poly-weave sacks.

- The sacks were closed with wire ties and taken at the end of each day to a locked shed in Fifield from where they were collected periodically for road transport to the laboratory. Sample submission forms were included.

In 2005, uniquely numbered plastic security tags were attached to each poly-weave sacks of samples at Fifield prior to despatch. Sample security forms were completed at Fifield and a copy was returned to the laboratory. These showed the seal number, drill hole number and number of samples for each sack. On receipt of the samples, the laboratory checked that the seal on each sack was intact, and that it contained the correct samples. The laboratory then marked and initialled the form accordingly and returned a copy to Ivanplats. The security arrangements in 2005 were adequate.

## 11.2 Independent custody sampling

Two separate programs of independent custody sampling have been conducted.

### 11.2.1 1999

In 1999, SLA conducted a program that involved the following steps:

- Duplicate samples were independently collected from the reject of routine samples, while holes were in progress. A total of 204 samples were collected from five RC holes. The duplicates were collected by separate personnel using a separate riffle splitter, under the direct supervision of SLA.
- The independent samples were immediately bagged and sealed, and then transported to Genalysis in Perth in the custody of SLA (initially in an SLA vehicle and then as airline cargo) before being delivered to the laboratory.

SLA reported that Nickel and cobalt results showed good correlation with the original samples (Section 11.6.2). The average grades reported were also similar to those from holes elsewhere in the deposit. This indicated that systematic tampering with routine samples was unlikely to have occurred.

### 11.2.2 2005

In 2005, McDonald Speijers conducted another, similar program. A total of 149 independent 1 m samples were collected, as shown in Table 11-1.

The bagged reject from the routine samples was riffle split by multiple passes through a free-standing, single-stage riffle to ~1 kg, for air freight to Perth. This re-splitting was conducted by contract labour under McDonald Speijers' direct supervision, independently of routine sample collection.

**Table 11-1: Independent custody sampling – 2005**

| Hole number | Interval (m) | Comments |
|---|---|---|
| SKRC1084 | 0 - 34 | |
| SKRC1085 | 0 - 34 | |
| SKRC1086 | 0 - 43 | 43 - 46 not resampled |
| SKRC1087 | 0 - 38 | 38 - 40 not resampled |

Individual samples were placed in small plastic bags which were closed with a wire tie. Samples were then placed into large plastic bags and wrapped with tape. These were in turn placed into woven sacks. The sacks were closed with wire ties, sealed with tape and initialled.

A total of eight sealed sacks were transported to Perth by McDonald Speijers, initially in a McDonald Speijers hire vehicle and subsequently by air as personal baggage. All sacks arrived intact with the signed seals unbroken.

The samples were then delivered by McDonald Speijers to Genalysis in Maddington, Western Australia, for preparation and assaying as follows:

- The total sample was pulverised to 90% -75 µm, with mills cleaned before and after the batch with barren quartz washes.
- Assaying:
  - For Pt, Au: 50 g fire assay, Pb collection, ICP finish
  - For Ni, Co, Mg, Fe, Cr, Mn, Al, Ca, Sc, Zn, Cu and As: mixed 4-acid digest with inductively coupled plasma atomic emission spectroscopy (ICP-OES) determination, using procedures similar to those used in October 1999 for SLA's independent samples (Jobs 279.0/994826 to 279.0/994829)
  - For Si: peroxide fusion followed by ICP-OES.

All analytical results were reported directly to McDonald Speijers. The assay results correlated satisfactorily with those from the original samples.

## 11.3 Sample preparation

### 11.3.1 ALS (pre-1999)

No documentation was located regarding sample preparation procedure used at ALS. McDonald Speijers contacted Peter Donaghy at ALS, Orange, who worked in the laboratory at the relevant time. Mr Donaghy indicated that the procedure would almost certainly have involved pulverising the total received samples to nominal -75 µm using Labtech-ESSA LM5 mills.

### 11.3.2 Ultratrace

The Ultratrace procedures also involved pulverising the total sample to a nominal 90% -75 µm using Labtech-ESSA LM5 mills. As part of routine internal laboratory quality control procedures, approximately 1 in 120 pulps were screened. The results were made available to Ivanplats. The average across 58 pulps was 94% passing 75 µm, with a minimum of 89%. These results were satisfactory.

## 11.4 Assaying

### 11.4.1 ALS (pre-1999)

Old laboratory reports and enquiries with ALS indicated that ALS' assaying procedures would have been as detailed in the following sections.

**Pt**

Fire assay using a 50 g charge, with Pb collection and an atomic absorption spectroscopy (AAS) finish (method PM217).

McDonald Speijers found mention of a very limited amount of screen fire assaying for Pt in reports dating from 1989. This involved no more than 30 samples, mostly quite low grade, and because of this, it was not useful in checking the extent to which Pt assays might be affected by coarse particles.

**Au**

The drill hole database indicated that Au assays from this period were obtained by Aqua Regia digest and AAS determination. However, it would seem more logical for the Au assays to have been obtained as part of the fire assay process. McDonald Speijers suspects that the database is misleading in this regard. Au values are generally very low and this is not a significant matter.

**Ni, Co, Cu, Cr, Mn**

These elements were determined using a perchloric acid digest of a 0.25 g portion of sample pulp, with an AAS finish (method G001).

Ni, Co and Cr were assayed routinely with Mn values for most samples from aircore holes. Only a very small number of Cu assays are recorded.

**Mg, Ca, Fe**

It appears that selected sample pulps were assayed retrospectively for these elements. A 0.25 g portion of pulp was digested in Aqua Regia with and ICP-OES finish (method IC581).

**Other elements**

The database contains very small numbers of assays (ranging only from 59 to 77) for Sc, Al and Si that are identified as originating from ALS. These mainly came from diamond drill holes SDD4 and SDD5. McDonald Speijers did not locate any information about the analytical method(s) used for these elements.

**Comments**

Apparently, at the time ALS did not consider the G001 method to be the most appropriate, but it was specified by the client (presumably for cost reasons).

### 11.4.2 Ultratrace (pre-2005)

Assay procedures used by Ultratrace prior to 2005 were as detailed in the following sections.

**Pt, Au**

Digestion of a nominal 40 g pulp portion in Aqua Regia, with an inductively coupled plasma mass spectrometry (ICPMS) finish.

Aqua Regia digests are often used to assay for precious metals in non-refractory mineralisation. However, there is a risk that the precious metals will not be totally digested. It is more common to use fire assay techniques when dealing with precious metal values of potential economic significance (as is the case with Pt in parts of this deposit).

**Ni, Co, Cr, Mn, Mg, Ca, Al, Fe, Sc, Zn, As, Cu**

Digestion to dryness of 0.3 g of sample pulp was done in a mixture of hot hydrochloric, nitric, perchloric and hydrofluoric acids. The residue was dissolved in hydrochloric acid, with an ICPOES finish. This method should have been generally suitable for the principal elements of economic interest (Nickel and cobalt).

**Moisture**

The moisture contents were routinely reported; these were based on the as received ("wet") weight and the weight after oven drying.

The samples involved were overwhelmingly from RC holes. Their moisture contents would have been reduced by the drying effects of large air volumes circulating through the holes and the time lag between drilling and delivery to the laboratory (apparently around 10 days). McDonald Speijers considers that reported moisture values are unlikely to provide a reliable indication of in situ moisture contents and McDonald Speijers would not include the moisture values in a resource model.

### 11.4.3 Ultratrace (2005)

Analytical procedures used by Ultratrace in 2005 were the same as those used in the 1998 - 2000 period, with the following exceptions:

- Pt and Au were determined by fire assay rather than Aqua Regia digest. The method used was normal Pb collection fire assaying of a 40 g charge with an ICP-OES finish.
- Moisture contents were not routinely determined.

In principle, assay results obtained in 2005 should have been consistent with earlier Ultratrace results for all elements, with the possible exception of Pt and Au.

### 11.4.4 ALS 2014 – 2015

For the 2014 - 2015 analysis, an aliquot of 0.25 g was digested in a mixture of perchloric, nitric, hydrofluoric and hydrochloric acids, and analysed for scandium and 32 other elements, including nickel and cobalt, by ICP-AES.

### 11.4.5 Comparison between assaying methods

A substantial number of aircore and RC holes originally assayed at ALS were subsequently re-assayed at Ultratrace. In effect, this provided a substantial set of inter-laboratory check assay data and it is discussed in Section 11.6.2.

The different analytical methods used at ALS (perchloric digest, AAS finish) and at Ultratrace (4-acid digest, ICP-OES finish) produced very similar results for nickel and cobalt. Only Cr values showed a substantial difference (Section11.6.2).

## 11.5 Sampling and assaying quality control procedures

The laboratories used would have had the normal internal quality control (QC) systems expected for reputable, commercial laboratories. McDonald Speijers has not reviewed internal laboratory QC results. External QC checks, preferably totally blind to the laboratory, carry more weight and McDonald Speijers has concentrated on these.

Very little information was available about external QC procedures prior to 1999.

However, most of the data used for resource estimation were generated since then, when external QC procedures have involved the use of:

- Field duplicate samples
- Check assaying at an independent laboratory
- Use of externally certified standard samples.

Procedures in 2005 were similar to those used in 1999 - 2000, but because of the substantial time gap, results from these two periods are discussed separately.

### 11.5.1 Duplicates

Duplicate samples were submitted to the primary laboratory at the rate of one per hole or roughly 1 in 35 samples. In all cases, they were collected in the field by riffle splitting the total reject from a selected sample in the mineralised zone. The samples involved were selected at the time by the geologists on the rigs.

According to standard operating procedures, duplicates were submitted to the laboratory in the same batch as the original sample. They were marked as a duplicate and with the hole number, but not with the sample depth, so the laboratory could not match the duplicate sample with the original sample.

The duplicates were appropriate for determining combined sub-sampling and assaying precision levels and for detecting any systematic bias in splitting procedures.

### 11.5.2 Check assaying

Check assaying was not done on a regular, routine basis, but in batches, usually towards the end of drilling programs.

In September 1999, 785 sample pulps were retrieved from Ultratrace and submitted to Genalysis in Maddington, Western Australia, for check assaying for Ni, Co, Cu, Fe, Mn, Cr, Sc, Ca, Al, Mg and Zn using a 4-acid digest with an ICP-OES finish (similar to the analytical method used by Ultratrace).

According to laboratory reports, the samples were submitted in a single batch on the same day, with all results reported over a two-day period in October 1999. This was not ideal. Check assaying should involve multiple batches spread over the duration of the drilling program in order to avoid the possibility of results being distorted by a transient assaying problem.

In 2005, a total of 231 pulps from routine samples were submitted to Genalysis for check assaying for the same suite of elements using the same method as in 1999. These were submitted in several batches, but laboratory reports indicate that they were all assayed on the same day.

In addition, another large set of older check assay data is provided by the substantial number of aircore and RC holes that were initially assayed by ALS between 1995 and 1998, with pulps from mineralised intervals subsequently being re-assayed by Ultratrace in 1999 or around that time.

No check assaying was done in 2014 - 2015.

### 11.5.3 Standards

Records indicate that sometime prior to the 1999 drilling programs, a total of five standard samples were prepared from stored aircore sample rejects. Unfortunately, the specific drill holes and intervals involved could not be identified. However, this would not compromise the use of the material as standards, provided that adequate assaying round robins were conducted in order to establish expected values and standard deviations.

Material from the selected intervals was apparently sent to Gannet Holdings Pty Ltd (Gannett) in Perth, Western Australia for bulk pulverising and homogenisation. McDonald Speijers understand that the final products were packaged into 25 g sealed sachets, ready for use.

Gannett apparently provided recommended Ni, Co, Mn, Fe, Mg and Al values for each standard on the basis of a round robin involving four sets of analyses for each standard from seven different laboratories in Australia. On the face of it, this should have been sufficient data to establish reasonably reliable expected values. McDonald Speijers obtained what appeared to be the original sets of assay data from which Gannett's recommended values were derived. These did not show any reasons why the expected values recommended by Gannett should not be accepted.

The standard numbers and their recommended values are shown in Table 11-2; these were extracted from a table (tblDH standards) in the drill hole database supplied to SRK (Syerston_DB.mdb). Each of these standards was apparently submitted under four different field standard identification numbers (Table 11-2).

In 2005, prior to the infill drilling program, remaining stocks of the old standards could not be located, so a total of five commercial standards derived from lateritic Ni mineralisation were obtained from Geostats Pty Ltd, White Gum Valley, Western Australia (Table 11-2). Recommended Nickel and cobalt values for these standards were based on between 50 and 118 assays from up to 50 - 60 laboratories worldwide.

There were no recommended values for Pt, so none of the standards was able to provide any check on the accuracy of Pt assays.

At the last minute, stocks of the old standards were located and these were also used in 2005.

In 2014 – 2015, only a single standard sample was used; this was intended primarily for monitoring Sc assay results. Nickel and cobalt grades of that standard were too low to provide useful data.

**Table 11-2: Standard sample recommended values and provenance**

| Standard Number | Field Standard Number | Prepared by | Provenance | Ni (ppm) | Co (ppm) |
|---|---|---|---|---|---|
| SYS1 | S4, S8, S12, S20 | Gannett | Syerston mineralisation | 6260 | 740 |
| SYS2 | S1, S6, S10, S16 | Gannett | Syerston mineralisation | 8980 | 2150 |
| SYS3 | S11, S14, S15, S19 | Gannett | Syerston mineralisation | 10120 | 3980 |
| SYS4 | S3, S7, S13, S17 | Gannett | Syerston mineralisation | 7480 | 2660 |
| SYS5 | S2, S8, S9, S18 | Gannett | Syerston mineralisation | 9330 | 3540 |
| GBM902-2 | GBM902-2 | Geostats Pty Ltd | Ni laterite ore, Queensland | 3014 | 975 |
| GBM901-1 | GBM901-1 | Geostats Pty Ltd | Nickel laterite, Kalgoorlie Region | 8037 | 1346 |
| GBM901-2C | GBM901-2C | Geostats Pty Ltd | Nickel laterite ore, Eastern Goldfields | 8830 | 314 |
| GBM302-8 | GBM302-8 | Geostats Pty Ltd | Nickel laterite, Eastern Goldfields | 10775 | 483 |
| GBM900-9 | GBM900-9 | Geostats Pty Ltd | Nickel laterite, Western Australia | 11615 | 567 |

Unfortunately, McDonald Speijers understands that in the 1999 - 2000 period, rather than being inserted in the field by Black Range, the external standards were supplied to the laboratory who inserted them into the sample batches as they were received. This was not good practice and it would have compromised the value of the results, because the expected values would obviously have been known to the laboratory.

In 2005, because they were located too late to be sent to site ahead of the drilling program, the old standards were again inserted at the laboratory. However, the new commercial standards were inserted in the field so that their identity was not known to the laboratory.

## 11.6 Sampling and assaying quality control results

### 11.6.1 Duplicates

Neither the original sample assay nor the duplicate result can be regarded as being inherently more reliable than the other. Consequently, following normal conventions, the relative difference of the original result from the mean of the two results has been used as a measure of the variation between the two sets of values. Mean relative differences should be close to zero if sub-sampling procedures are free of bias. The standard deviation of relative differences provides a basis for calculating combined sub-sampling and assaying precision levels.

Precision levels can be extremely sensitive to the impact of a small number of erratic, outlier values. These are often due to simple human errors such as sample interchange, transcription errors or decimal point errors, that do not reflect real levels of sub-sampling and assaying error (although with precious metals some can be real and related to coarse particulate metal). McDonald Speijers therefore identified probable outliers from histograms of relative differences and excluded these from final statistics. The proportions of outliers were typically 1% - 3%, within the normal range that McDonald Speijers observes for similar datasets.

## 1999 – 2000 results

McDonald Speijers extracted results for 619 duplicates from the table "tblDHduplicateassays" in the old Black Range drill hole database (Syerston_DB.mdb). Most of the assays for duplicate samples were listed in columns identifying them as Genalysis assays. However, spot checks against original laboratory reports indicated that they were actually from Ultratrace, as specified in the standard operating procedures of the time.

Summary statistics for duplicate samples submitted to Ultratrace in 1999 - 2000 are shown in Table 11-3.

## 2005 results

A total of 117 duplicate samples were collected and assayed during the 2005 drilling program.

Results were similar to those reported from 1999 - 2000, except that the proportion of outliers in the results for Co and for Mg were considerable higher than normal. The results are summarised in Table 11-4, Figure 11-1 and Figure 11-2.

## Discussion

Results for Ni were consistently satisfactory. There was no evidence of sub-sampling bias in either dataset. Precision levels were well within the range normally considered to be satisfactory for base metals (about ±10% at a 90% confidence level) and they were similar for both sets of results.

Results were satisfactory for Co, although the proportion of outliers in the datasets was higher, particularly in the 2005 results where it reached unusual levels. The reason for this is not known, but McDonald Speijers suspects that some interchange of samples or results might have occurred in 2005.

There were no indications of sub-sampling bias, and after the removal of probable outliers, precision levels for Co were within satisfactory limits and similar over time.

**Table 11-3: Duplicate samples, 1999 – 2000, summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Original | Duplicate | Mean | Relative difference | Proportion excluded | Original | Duplicate | Mean | Relative difference |
| Ni | Pairs | 619 | | | | 1.5% | 610 | | | |
| | Minimum | 0.011 | 0.013 | 0.012 | -35.0% | | 0.011 | 0.013 | 0.012 | -11.6% |
| | Maximum | 2.27 | 2.25 | 2.26 | 60.5% | | 2.27 | 2.25 | 2.26 | 16.1% |
| | Mean | 0.71 | 0.70 | 0.71 | 0.4% | | 0.71 | 0.71 | 0.71 | 0.1% |
| | Standard Deviation | 0.34 | 0.34 | 0.34 | 5.2% | | 0.34 | 0.34 | 0.34 | 2.3% |
| | Precision at 90% Confidence Limits | | | | 12.2% | | | | | 5.3% |
| Co | Pairs | 619 | | | | 2.3% | 605 | | | |
| | Minimum | 0.002 | 0.002 | 0.002 | -82.6% | | 0.002 | 0.002 | 0.002 | -18.0% |
| | Maximum | 0.702 | 0.692 | 0.697 | 82.8% | | 0.702 | 0.692 | 0.697 | 17.5% |
| | Mean | 0.13 | 0.13 | 0.13 | 0.5% | | 0.13 | 0.13 | 0.13 | -0.1% |
| | Standard Deviation | 0.12 | 0.12 | 0.12 | 9.3% | | 0.12 | 0.12 | 0.12 | 3.0% |
| | Precision at 90% Confidence Limits | | | | 21.7% | | | | | 7.0% |

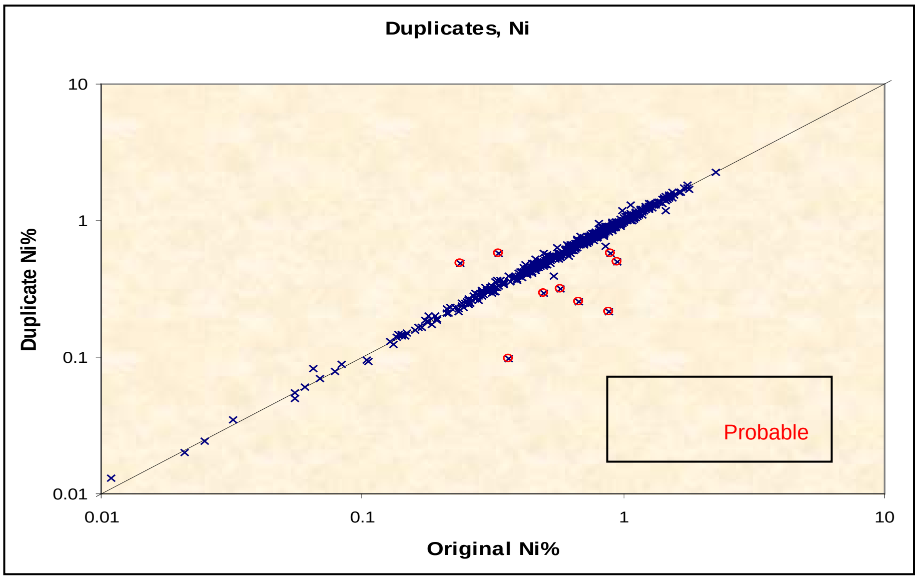



**Figure 11-1: Duplicate samples, 1999 - 2000 (Ni)**

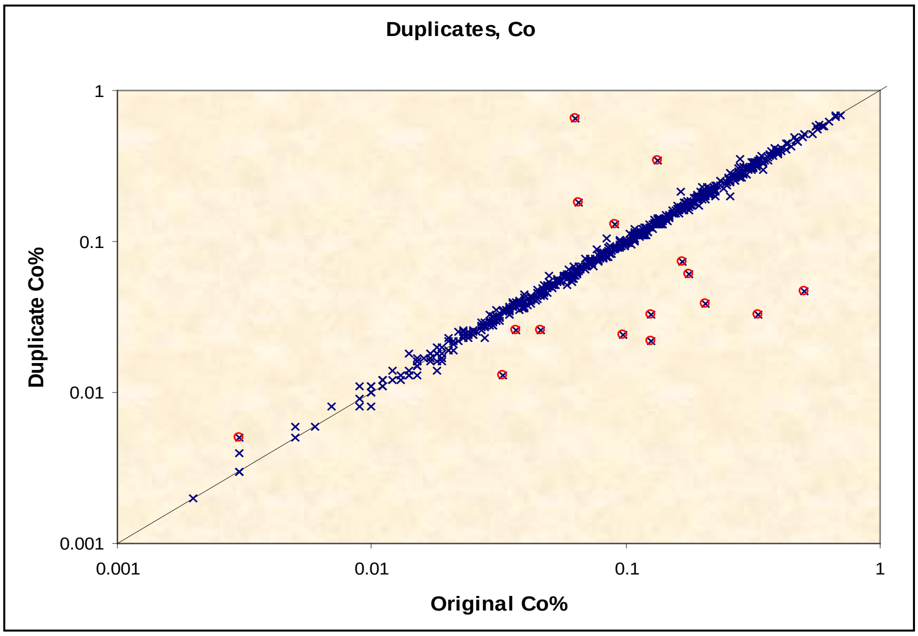



**Figure 11-2: Duplicate samples, 1999 - 2000 (Co)**

**Table 11-4: Duplicate samples, 2005, summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Original | Duplicate | Mean | Relative difference | Proportion excluded | Original | Duplicate | Mean | Relative difference |
| Ni | Pairs | 117 | 117 | 117 | 117 | 1.7% | 115 | 115 | 115 | 115 |
| | Minimum | 0.071 | 0.072 | 0.0715 | -25.6% | | 0.071 | 0.072 | 0.0715 | -10.7% |
| | Maximum | 1.67 | 1.63 | 1.65 | 81.1% | | 1.67 | 1.63 | 1.65 | 8.5% |
| | Mean | 0.78 | 0.77 | 0.78 | 0.9% | | 0.78 | 0.77 | 0.78 | 0.5% |
| | Standard Deviation | 0.29 | 0.29 | 0.28 | 8.3% | | 0.29 | 0.28 | 0.28 | 2.6% |
| | Precision (at 90% CL) | | | | 19.2% | | | | | 6.1% |
| Co | Pairs | 117 | 117 | 117 | 117 | 6.0% | 110 | 110 | 110 | 110 |
| | Minimum | 0.011 | 0.009 | 0.0115 | -83.0% | | 0.011 | 0.012 | 0.0115 | -8.5% |
| | Maximum | 0.878 | 0.84 | 0.859 | 93.7% | | 0.878 | 0.84 | 0.859 | 10.9% |
| | Mean | 0.23 | 0.23 | 0.23 | 0.4% | | 0.23 | 0.23 | 0.23 | 0.6% |
| | Standard Deviation | 0.18 | 0.18 | 0.18 | 15.3% | | 0.18 | 0.18 | 0.18 | 3.6% |

Precision levels for other major elements were in the range of about ±7% - 20%.

The worst precision limits were for Mg and Mn in 2005 which were around ±14% and ±20% respectively. Otherwise, they were generally within about ±10% - 12%, which should be adequate.

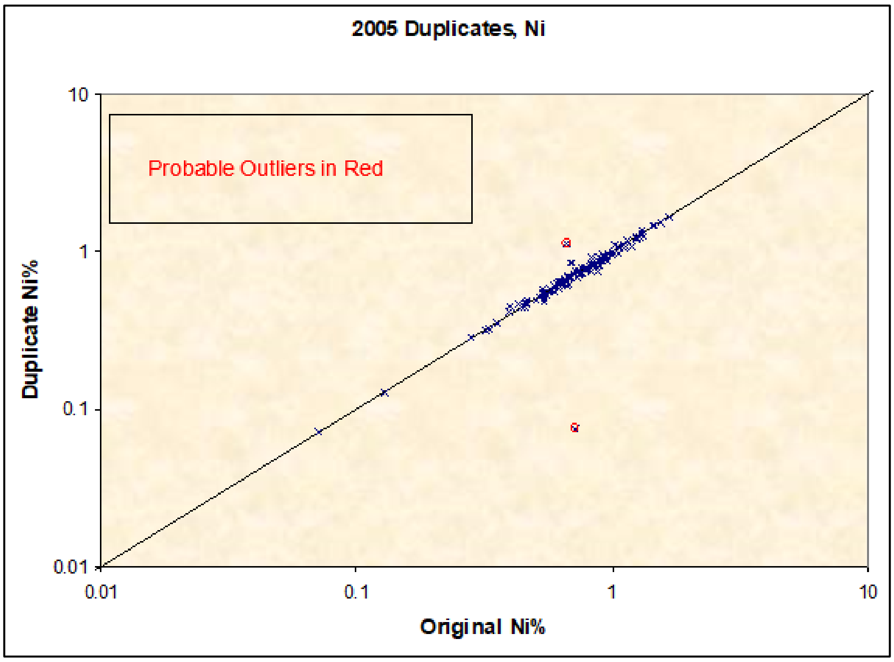



**Figure 11-3: Duplicate samples, 2005 (Ni)**

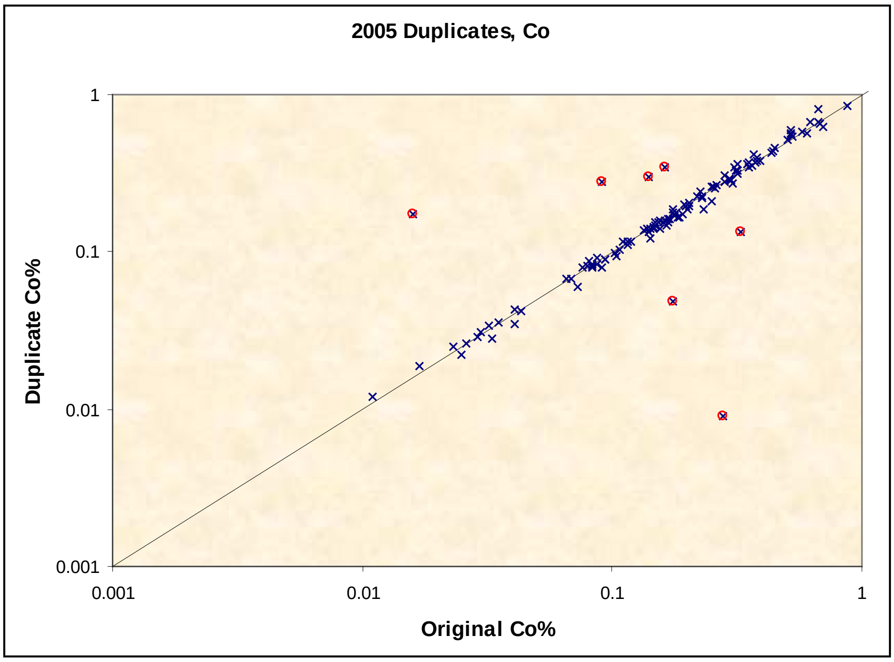



**Figure 11-4: Duplicate samples, 2005 (Co)**

## 2014 - 2015

McDonald Speijers obtained results compiled by OreWin for 17 duplicate samples from 2 m intervals and 88 duplicate samples from 1 m intervals in RC holes drilled in 2014 - 2015. This equated to a rate of about 1 in 25 samples.

Taken as a whole, the results showed noticeably poorer precision than previous sets of field duplicate samples and a tendency for the duplicates to give slightly higher results for both Nickel and cobalt.

However, the apparent biases were restricted to very low grade samples (<0.05% Ni and <0.01% Co). When these were excluded, biases were no longer evident (Figure 11-5).

McDonald Speijers understands that duplicates were collected by spear sampling the bagged reject, although McDonald Speijers has not been able to independently verify this. If correct, this was an unacceptable procedure. A new set of valid duplicates should be obtained by riffle splitting the bagged reject.

A poor duplicate sampling procedure would explain the relatively poor apparent precision levels indicated by the results, which would have little meaning and would not provide an indication of real precision levels.

It would be highly advisable to use either Ultratrace or Genalysis as the check laboratory, specifying the same analytical method they used in 2005, in order to retain a link to assaying procedures during previous drilling programs that provided the bulk of the assay database.

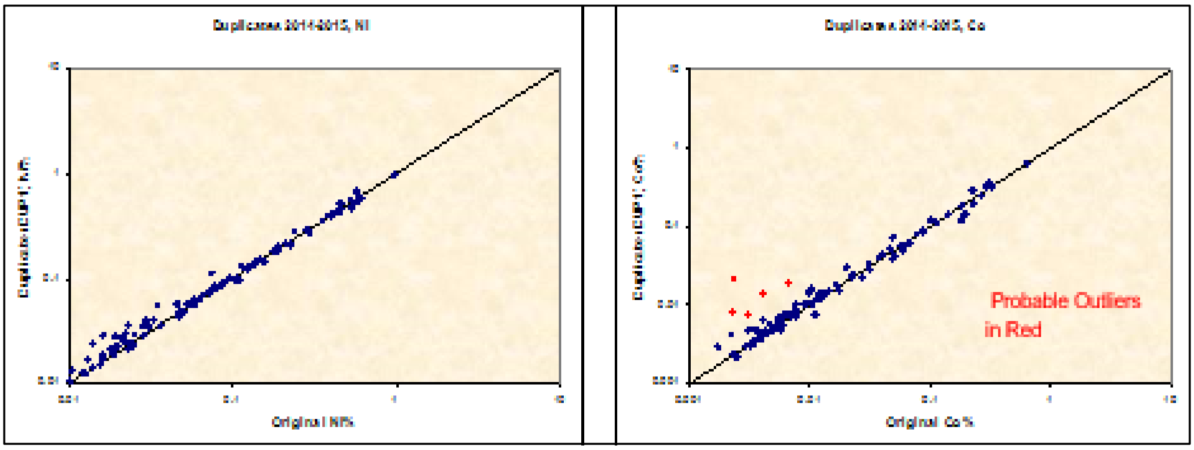



**Figure 11-5: Duplicate samples, 2014 - 2015 (Ni & Co)**

## 11.6.2 Check assays

Check assays were treated in a similar fashion to duplicate samples, since neither the original nor the check result could be assumed to be inherently more accurate than the other. The relative difference of the Ultratrace result from the mean of both results was calculated.

Mean relative differences provide a measure of any analytical bias between the laboratories. Ideally, these should be near zero, but in SRK's experience, the mean relative differences between reputable laboratories may fall in the range of about ±5%, although they should preferably be within ±2% - 3%.

### Pre-1999 results

McDonald Speijers extracted 4,491 samples from the database that had Ni assays from both ALS and Ultratrace and treated these as effectively being a large check assay dataset. There were similar amounts of data for Co, Mg, Cr and Mn, more limited data for Fe, Mg and Ca, and small datasets for a few other elements. There was no comparative assay information for Pt. The results from this period are summarised in Table 11-5.

### 1999 – 2000 results

McDonald Speijers obtained results for 785 samples assayed by both Ultratrace and Genalysis for Ni, Co, Cu, Fe, Mn, Cr, Sc, Ca, Al, Mg and Zn in the 1999 - 2000 period. These were contained in a spreadsheet (Genalysis Checks_Graphs.xls) in an old SLA archive directory provided to SRK by Ivanplats. McDonald Speijers spot checked some of the results against original Genalysis laboratory reports and confirmed that the results were valid.

This dataset also failed to provide any quality control on Pt assays. Summary statistics for this period are shown in Table 11-6.

### 2005 results

In 2005, a total of 231 original sample pulps were retrieved from Ultratrace and submitted to Genalysis for check assaying for Ni, Co, Cu, Fe, Mn, Cr, Ca, Al, Mg and Zn.

The results are summarised in Figure 11-6 and Figure 11-7.

### Discussion

All the available check assay results for Ni were very good and indicated that Ni assays have been accurate over the entire time period involved. The results from Ultratrace compared well with those from ALS and with both generations of Genalysis checks, the mean relative differences are within about ±1% in all cases.

**Table 11-5: Check assays (pre-1999) – ALS vs Ultratrace – summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | ALS | Ultratrace | Mean | Relative difference | Proportion excluded | ALS | Ultratrace | Mean | Relative difference |
| Ni% | Pairs | 4491 | 4491 | 4491 | 4491 | 0.3% | 4476 | 4476 | 4476 | 4476 |
| | Minimum | 0.003 | 0.0019 | 0.00245 | -36.4% | | 0.003 | 0.0019 | 0.00245 | -12.4% |
| | Maximum | 2.39 | 2.82 | 2.605 | 82.2% | | 2.39 | 2.82 | 2.605 | 22.4% |
| | Mean | 0.55 | 0.55 | 0.55 | 1.1% | | 0.55 | 0.55 | 0.55 | 1.0% |
| | Standard Deviation | 0.28 | 0.30 | 0.29 | 5.3% | | 0.28 | 0.29 | 0.29 | 4.3% |
| Co% | Pairs | 4448 | 4448 | 4448 | 4448 | 0.2% | 4439 | 4439 | 4439 | 4439 |
| | Minimum | 0.001 | 0.0017 | 0.00135 | -95.6% | | 0.001 | 0.0017 | 0.00135 | -34.6% |
| | Maximum | 2.19 | 2.13 | 2.16 | 82.5% | | 2.19 | 2.13 | 2.16 | 27.5% |
| | Mean | 0.093 | 0.089 | 0.091 | 2.1% | | 0.093 | 0.089 | 0.091 | 2.1% |
| | Standard Deviation | 0.109 | 0.107 | 0.108 | 6.5% | | 0.109 | 0.106 | 0.107 | 5.7% |

**Table 11-6: Check assays (1999 – 2000) – Genalysis vs Ultratrace – summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Ultratrace | Genalysis | Mean | Relative difference | Proportion excluded | Ultratrace | Genalysis | Mean | Relative difference |
| Ni% | Pairs | 785 | 785 | 785 | 785 | 0.0% | 785 | 785 | 785 | 785 |
| | Minimum | 0.184 | 0.195 | 0.192 | -11.5% | | 0.184 | 0.195 | 0.192 | -11.5% |
| | Maximum | 2.22 | 2.15 | 2.185 | 10.4% | | 2.22 | 2.15 | 2.185 | 10.4% |
| | Mean | 0.76 | 0.76 | 0.76 | -0.1% | | 0.76 | 0.76 | 0.76 | -0.1% |
| | Standard Deviation | 0.34 | 0.33 | 0.33 | 2.8% | | 0.34 | 0.33 | 0.33 | 2.8% |
| Co% | Pairs | 785 | 785 | 785 | 785 | 0.5% | 781 | 781 | 781 | 781 |
| | Minimum | 0.005 | 0.005 | 0.005 | -59.2% | | 0.005 | 0.005 | 0.005 | -20.1% |
| | Maximum | 1.04 | 1.00 | 1.02 | 59.2% | | 1.04 | 1.00 | 1.020 | 13.3% |
| | Mean | 0.12 | 0.12 | 0.12 | -0.9% | | 0.12 | 0.12 | 0.12 | -0.9% |
| | Standard Deviation | 0.14 | 0.13 | 0.14 | 5.2% | | 0.14 | 0.13 | 0.14 | 4.0% |

**Table 11-7: Check assays (2005) – Genalysis vs Ultratrace – summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Ultratrace | Genalysis | Mean | Relative difference | Proportion excluded | Ultratrace | Genalysis | Mean | Relative difference |
| Ni% | Pairs | 231 | 231 | 231 | 231 | 0.0% | 231 | 231 | 231 | 231 |
| | Minimum | 0.19 | 0.19 | 0.19 | -7.0% | | 0.19 | 0.19 | 0.19 | -7.0% |
| | Maximum | 1.59 | 1.82 | 1.70 | 5.9% | | 1.59 | 1.82 | 1.70 | 5.9% |
| | Mean | 0.75 | 0.76 | 0.75 | -0.3% | | 0.75 | 0.76 | 0.75 | -0.3% |
| | Standard Deviation | 0.27 | 0.28 | 0.27 | 2.6% | | 0.27 | 0.28 | 0.27 | 2.6% |
| Co% | Pairs | 231 | 231 | 231 | 231 | 0.4% | 230 | 230 | 230 | 230 |
| | Minimum | 0.008 | 0.009 | 0.008 | -10.4% | | 0.008 | 0.009 | 0.008 | -10.4% |
| | Maximum | 1.230 | 1.265 | 1.248 | 81.0% | | 1.230 | 1.265 | 1.248 | 5.8% |
| | Mean | 0.164 | 0.167 | 0.166 | -1.2% | | 0.163 | 0.168 | 0.165 | -1.6% |
| | Standard Deviation | 0.196 | 0.203 | 0.199 | 6.2% | | 0.195 | 0.204 | 0.199 | 3.0% |

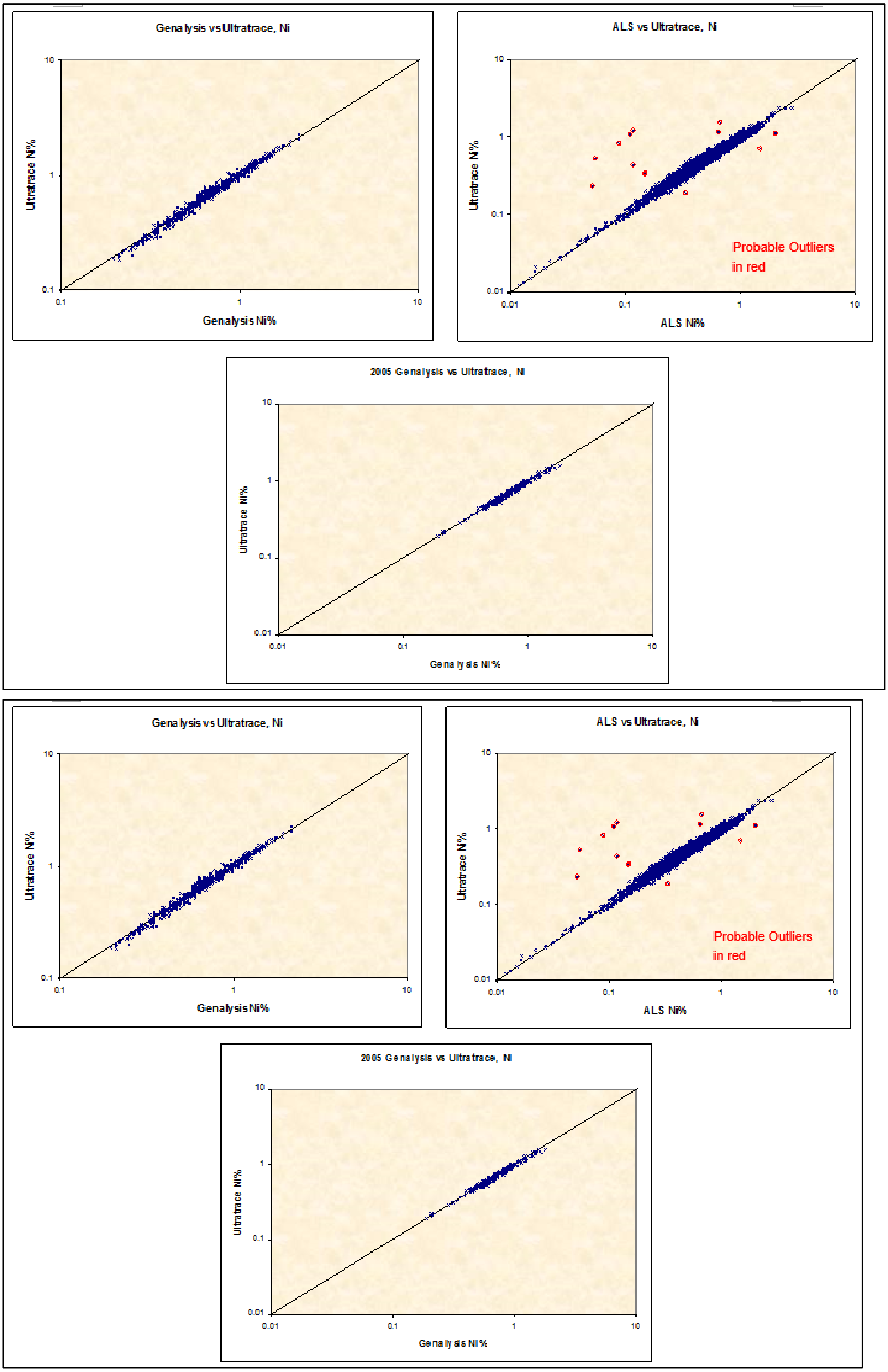
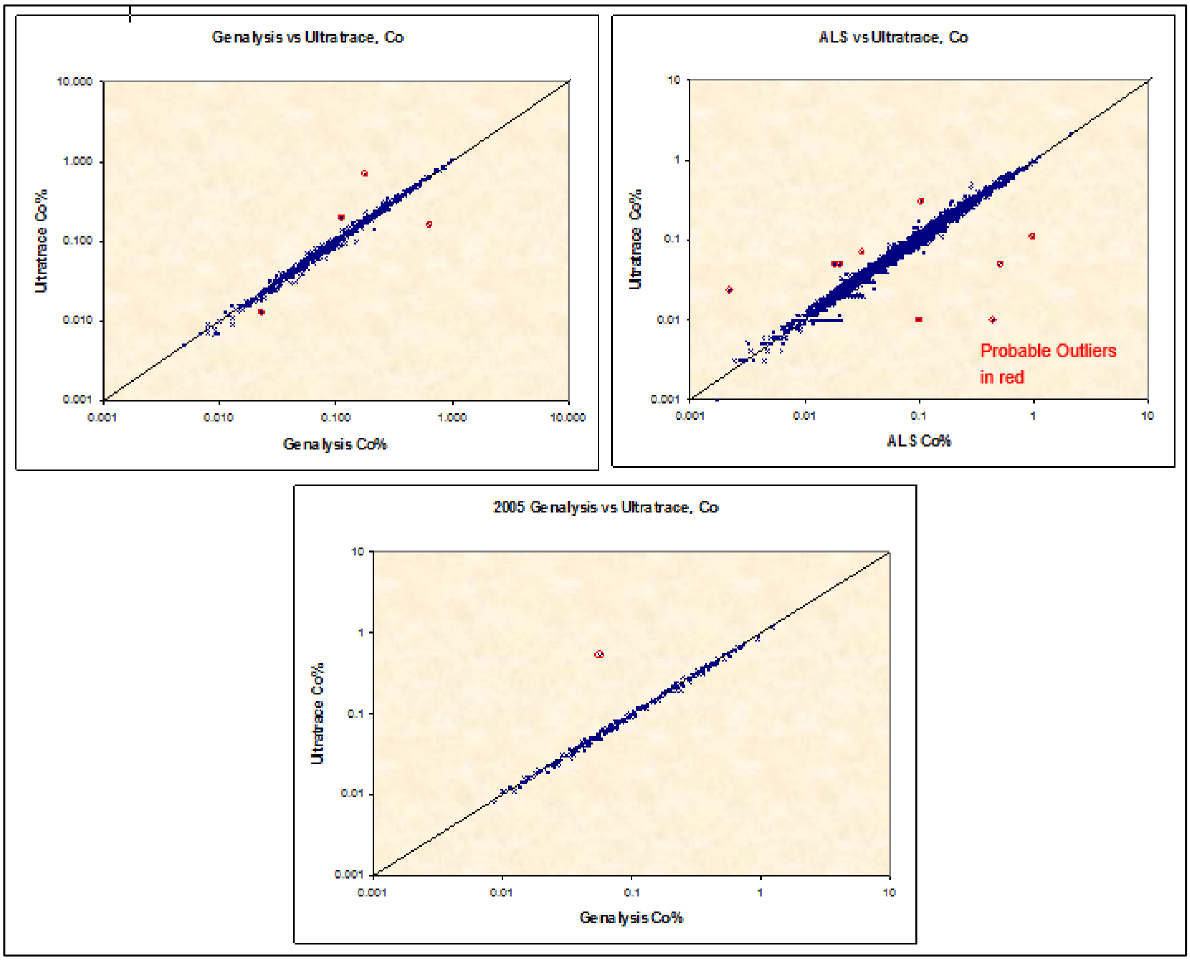



**Figure 11-6: Scatterplots, check assays (Ni)**

Check assay results for Co were also good in all cases with mean relative differences not exceeding about 2%. On average, Ultratrace tended to be slightly higher than ALS and slightly lower than Genalysis.

Overall, the check assay results indicated that Co assays have been accurate (Figure 11-7).



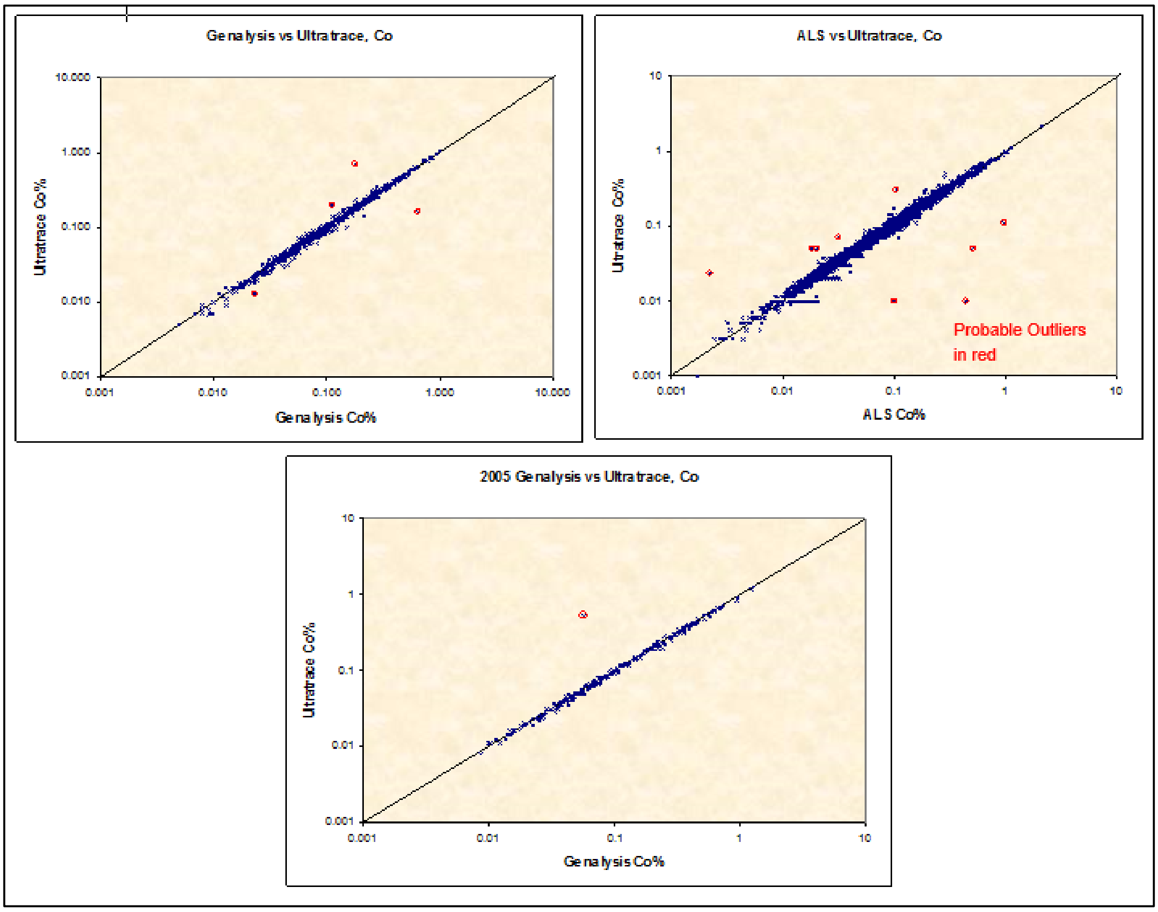



**Figure 11-7: Check assays (Co)**

The most significant difference was with ALS; this was due to a small proportion of values that compared very poorly, while most others compared well. At this time, McDonald Speijers was unable to identify any common factors linking the poor results, e.g. particular assay batches or a discrete part of the deposit.

### 11.6.3 Independent custody samples

#### SLA (1999)

In 1999, SLA submitted 204 independent custody samples to Genalysis for assay by ICP-OES. These were duplicate field splits of routine samples. They effectively constituted another set of check assays, although because the analyses were conducted on duplicate splits rather than the same laboratory pulps, a greater amount of variability would be expected.

McDonald Speijers was unable to locate digital data for these samples.

SLA reported that results for nickel and cobalt agreed very well with the routine Ultratrace values (Table 11-8), but because McDonald Speijers did not have original digital data McDonald Speijers could not calculate relative difference statistics or generate scatterplots.

McDonald Speijers also noted that Cr values correlated poorly and there was potential under-reporting of Cr grades by Ultratrace. This was consistent with other check assay results.

Ultratrace tended to give lower Al values above about 0.8%.

Si values correlated poorly. The Genalysis Si assays were all by peroxide fusion and McDonald Speijers already noted that Ultratrace fusion assays have shown that there are inaccuracies in the routine calculated Si values.

**Table 11-8: Summary statistics of SLA custody sampling for nickel and cobalt**

| | Ni ppm | | Co ppm | |
|---|---|---|---|---|
| | **Genalysis (SLA)** | **Ultratrace (Black Range)** | **Genalysis (SLA)** | **Ultratrace (Black Range)** |
| Count | 204 | 204 | 204 | 204 |
| Minimum | 145 | 120 | 6 | 10 |
| Maximum | 17000 | 17000 | 5600 | 5650 |
| Mean | 7113 | 6989 | 843 | 830 |
| Standard Deviation | 4661 | 4642 | 1039 | 1039 |
| Correlation Coefficient | 0.9966 | | 0.9971 | |

Source: SLA, July 2000.

## McDonald Speijers (2005)

In 2005, McDonald Speijers submitted 149 independent custody samples to Genalysis. In two cases, the first four original samples from a hole had been composited prior to submission for assay. The 1 m independent sample results for these intervals were consequently averaged and compared with the original 4 m composite sample results. In one case, the original samples from the first 4 m of a hole were logged as transported material and not submitted for assay. This left 139 pairs of assay results available for statistical analysis. The results are summarised in Table 11-9.

**Table 11-9: 2005 independent custody samples – summary statistics**

| Element | | All data | | | | Probable outliers excluded | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Ultratrace | Genalysis | Mean | Relative difference | Proportion excluded | Ultratrace | Genalysis | Mean | Relative difference |
| Ni% | Pairs | 139 | 139.000 | 139 | 139 | 0.0% | 139 | 139.000 | 139 | 139 |
| | Minimum | 0.05 | 0.04 | 0.04 | -5.9% | | 0.05 | 0.04 | 0.04 | -5.9% |
| | Maximum | 0.84 | 0.85 | 0.83 | 13.9% | | 0.84 | 0.85 | 0.83 | 13.9% |
| | Mean | 0.41 | 0.40 | 0.40 | 1.7% | | 0.41 | 0.40 | 0.40 | 1.7% |
| | Standard Deviation | 0.23 | 0.22 | 0.22 | 3.8% | | 0.23 | 0.22 | 0.22 | 3.8% |
| Co% | Pairs | 139 | 139 | 139 | 139 | 0.0% | 139 | 139 | 139 | 139 |
| | Minimum | 0.006 | 0.005 | 0.005 | -7.2% | | 0.006 | 0.005 | 0.005 | -7.2% |
| | Maximum | 0.631 | 0.579 | 0.605 | 12.0% | | 0.631 | 0.579 | 0.605 | 12.0% |
| | Mean | 0.100 | 0.095 | 0.097 | 1.8% | | 0.100 | 0.095 | 0.097 | 1.8% |
| | Standard Deviation | 0.109 | 0.104 | 0.106 | 4.2% | | 0.109 | 0.104 | 0.106 | 4.2% |

Results for Ni compared satisfactorily (Figure 11-8). Ultratrace tended to be slightly higher, with a mean relative difference of only about 2%.

Results for Co were also satisfactory (Figure 11-8). Ultratrace again tended to be slightly higher, with the mean relative difference being approximately 2%. Taken together, the Nickel and cobalt results were quite satisfactory and indicated that the original samples had not been tampered with. The comparison between the laboratories was similar to that indicated by the batches of normal check assays.

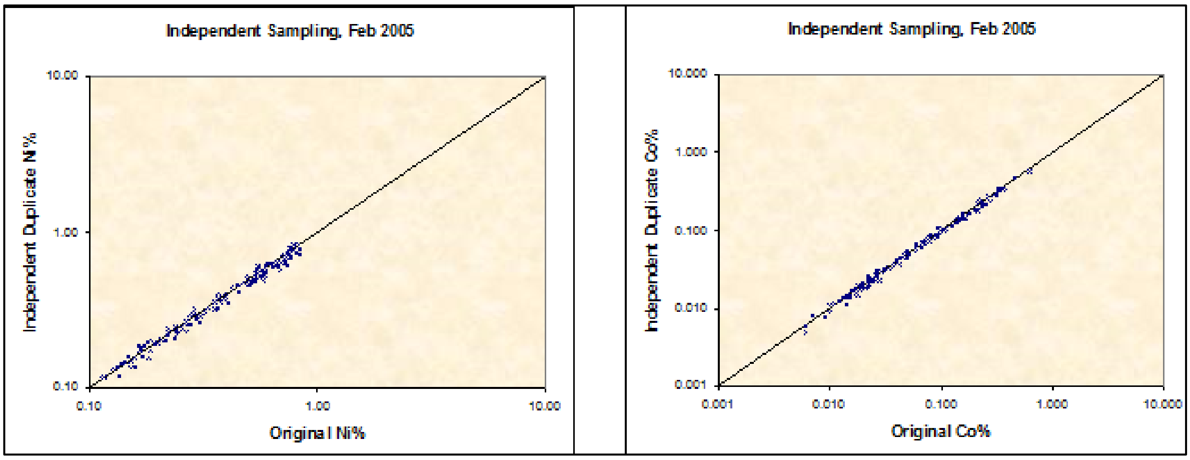



**Figure 11-8: Independent custody samples – scatterplots for Nickel and cobalt**

### Other elements

Results for Fe, Cu, Mg, Sc, and Zn compared satisfactorily (mean relative differences within about ±3%).

Results for Al were satisfactory for samples above about 1% Al. Below this, there were large relative differences, but these were of little practical significance.

Mn results from Ultratrace tended to be consistently slightly higher than those from the independent samples, with the mean relative difference being about 5%.

Ultratrace results for Ca also tended to be higher, with a mean relative difference of about 8%.

Consistent with all other inter-laboratory comparisons, Cr results from Ultratrace tended to be lower, the mean relative difference being about -5%.

Assays for Si involved different methods. A peroxide fusion method was used on the independent samples, while Si values for the original samples were calculated from other assay data using regression equations. The results were similar to other comparisons between these two methods, with the calculated values from Ultratrace tending to be only slightly higher lower for samples with low Fe contents (<30% Fe), but significantly lower for those with high Fe content.

## 11.6.4 Standards

When dealing with standard samples, McDonald Speijers has assumed that the recommended value is the best available estimate of the true grade. Relative differences for reported assays were therefore calculated using the recommended values as the base.

### 1999 – 2000 results

McDonald Speijers extracted 766 sets of Ultratrace assay results for standard samples from a table (tblDHStandardassays) in the drill hole database supplied in late 2004 (Syerston_DB.mdb).

Unfortunately, 680 of these did not have a valid standard number, being identified simply as "S".

McDonald Speijers was forced to deduce the standard on the basis of reasonably distinctive combinations of assay values for various elements as shown in Figure 11-9, e.g. SYS2, 3 & 5 could be distinguished on the basis of substantially different Al & Co values.

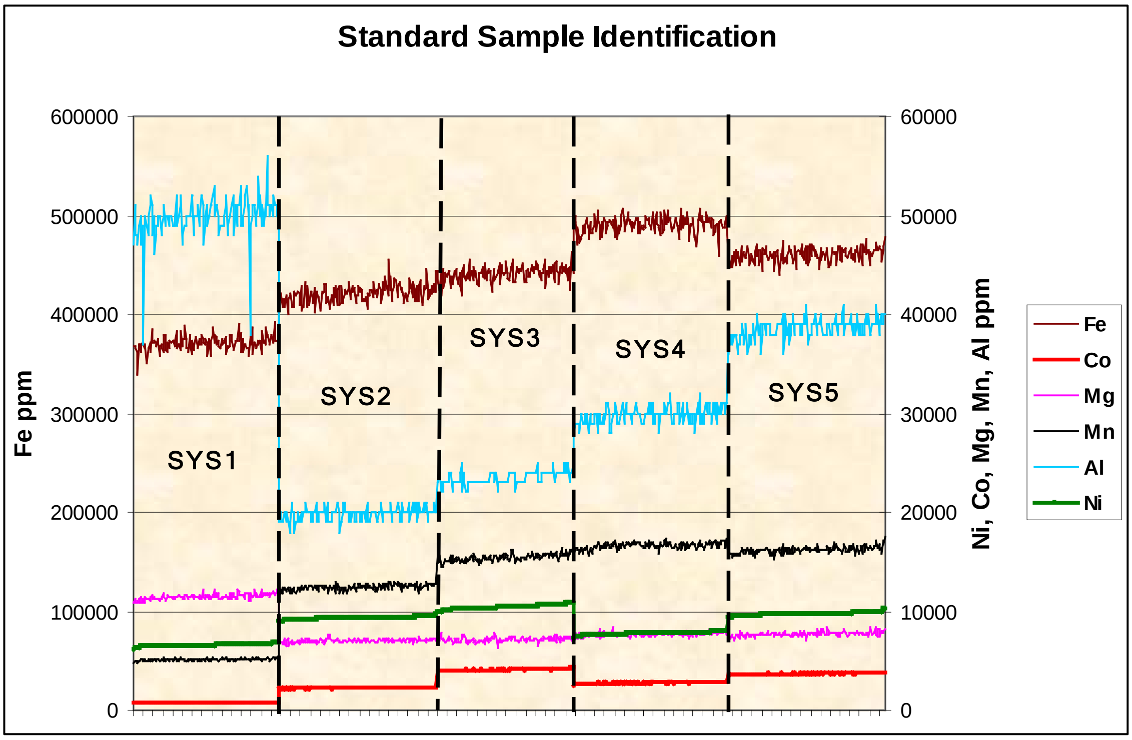



**Figure 11-9: Chart showing assumed standard sample identifications – 1999 - 2000**

There was a general tendency for Ultratrace to return slightly higher values than expected and it had apparently been suggested in the past that the recommended values for these standards might be somewhat too low. McDonald Speijers therefore collated what appeared to be the original Gannett round robin assay results for nickel and cobalt from old hard copy information. The number of results for each standard was not high (a total of 28, 4 from each of 7 laboratories) but there was no obvious reason why any of them should have been rejected and the average result from each of the laboratories involved lay within a ±5% relative band around the recommended values. There were no apparent reasons why any of the recommended values should be modified.

**Table 11-10: Standard samples – 1999 - 2000 – summary statistics**

| Assumed standard number | | Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% |
|---|---|---|---|---|---|---|
| SYS1 | Count | 148 | | 148 | 148 | |
| | Minimum | 0.602 | | -3.8% | 0.072 | |
| | Maximum | 0.682 | | 8.9% | 0.084 | |
| | Mean | 0.656 | 0.626 | 4.7% | 0.078 | 0.074 |
| | Standard Deviation | 0.012 | | 1.9% | 0.002 | |
| SYS2 | Count | 161 | | 161 | 161 | |
| | Minimum | 0.9 | | 0.2% | 0.218 | |
| | Maximum | 0.974 | | 8.5% | 0.233 | |
| | Mean | 0.930 | 0.898 | 3.6% | 0.225 | 0.215 |
| | Standard Deviation | 0.012 | | 1.4% | 0.003 | |

| Assumed standard number | | Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% |
|---|---|---|---|---|---|---|
| SYS3 | Count | 139 | | 139 | 139 | |
| | Minimum | 0.996 | | -1.6% | 0.39 | |
| | Maximum | 1.090 | | 7.7% | 0.435 | |
| | Mean | 1.041 | 1.012 | 2.9% | 0.410 | 0.398 |
| | Standard Deviation | 0.021 | | 2.1% | 0.008 | |
| SYS4 | Count | 157 | | 157 | 157 | |
| | Minimum | 0.724 | | -3.2% | 0.256 | |
| | Maximum | 0.809 | | 8.2% | 0.290 | |
| | Mean | 0.775 | 0.748 | 3.5% | 0.278 | 0.266 |
| | Standard Deviation | 0.014 | | 1.9% | 0.005 | |
| SYS5 | Count | 161 | | 161 | 161 | |
| | Minimum | 0.933 | | 0.0% | 0.351 | |
| | Maximum | 1.020 | | 9.3% | 0.390 | |
| | Mean | 0.972 | 0.933 | 4.1% | 0.370 | 0.354 |
| | Standard Deviation | 0.014 | | 1.5% | 0.007 | |

According to its 2000 report, SLA submitted a series of standard sample pulps to both Ultratrace and Genalysis, some being “baked” in drying ovens and some submitted directly in order to test the possibility that prolonged drying might have changed their chemistry.

SLA reported that results from the two laboratories for Nickel and cobalt were "almost identical" with no significant difference between "baked" and "unbaked" samples, but McDonald Speijers could not locate any detailed data. The reported result was consistent with check assay results that indicated no significant analytical bias between these laboratories for Ni or Co.

## 2005 results

Standard sample results were collated for laboratory assay batches up to and including SRC1193 (the last of the holes in the database available for use in resource estimation).

There was a total of 116 results for the five old Syerston standards and 104 for the five commercial standards. In 13 cases, it was evident from the assay results that the commercial standard inserted in the field had been incorrectly recorded. Because some of the standards involved had similar values, it was not always possible to confidently deduce which of them had actually been used, so these results were all discarded, leaving 91 accepted results for the commercial standards.

Control charts were prepared for all standards. These charts are used to monitor results for failures. A failure would normally be defined as a standard assay result falling outside a range of ±2.5 or ±3 standard deviations from the expected value. The failure level would normally be agreed between the client and the laboratory, and a failure would usually result in the entire batch concerned being re-assayed.

The following were the only potential failures, at around ±2.5 standard deviations:

- GBM901-1 (Nickel and cobalt): Batch u64333
- SYS3 (Ni): Batch u64316
- SYS5 (Ni): Batch u64333.

However, results for other external standards in the same assay batches as these lay within potential failure limits, so re-assaying of the batches involved could not be readily justified.

While there were no definite failures, the results for Nickel and cobalt showed a consistent bias, with the average assays tending to be higher than expected values (Table 11-11).

## Discussion

Assay results for nickel and cobalt from Ultratrace tended to be consistently higher than the expected values.

For the old Syerston standards, mean relative differences were typically about 3% - 5%, averaging about 3% - 4%. The results were very consistent between the 1999 - 2000 and 2005 datasets, as shown in Figure 11-10 and Figure 11-11. These differences were within generally acceptable limits of about ±5% relative.

However, for the commercial standards used in 2005, mean relative differences were appreciably larger, ranging from about 3% - 14% and typically about 8% (Figure 11-10 and Figure 11-11). Many of these were outside an acceptable range.

**Table 11-11: Standards results**

| Standard Number | | Ultratrace results Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% | Relative difference (Co) | Genalysis results Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% | Relative difference (Co) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| SYS1 | Count | 24 | | 24 | 24 | | 24 | 3 | | 3 | 3 | | 3 |
| | Minimum | 0.619 | | -1.1% | 0.071 | | -4.1% | 0.653 | | 4.3% | 0.078 | | 4.9% |
| | Maximum | 0.679 | | 8.5% | 0.079 | | 6.8% | 0.664 | | 6.1% | 0.080 | | 8.6% |
| | Mean | 0.649 | 0.626 | 3.6% | 0.075 | 0.074 | 1.8% | 0.658 | 0.626 | 5.1% | 0.079 | 0.074 | 6.3% |
| | Standard Deviation | 0.013 | | 2.1% | 0.002 | | 2.4% | 0.006 | | 0.9% | 0.002 | | 2.0% |
| SYS2 | Count | 23 | | 23 | 23 | | 23 | 3 | | 3 | 3 | | 3 |
| | Minimum | 0.908 | | 1.1% | 0.216 | | 0.5% | 0.927 | | 3.2% | 0.228 | | 6.1% |
| | Maximum | 0.967 | | 7.7% | 0.233 | | 8.4% | 0.964 | | 7.4% | 0.238 | | 10.7% |
| | Mean | 0.932 | 0.898 | 3.8% | 0.223 | 0.215 | 3.6% | 0.951 | 0.898 | 5.9% | 0.233 | 0.215 | 8.3% |
| | Standard Deviation | 0.014 | | 1.5% | 0.004 | | 2.1% | 0.021 | | 2.3% | 0.005 | | 2.3% |
| SYS3 | Count | 20 | | 20 | 20 | | 20 | 3 | | 3 | 3 | | 3 |
| | Minimum | 0.975 | | -3.7% | 0.385 | | -3.3% | 1.029 | | 1.7% | 0.412 | | 3.5% |
| | Maximum | 1.100 | | 8.7% | 0.427 | | 7.3% | 1.059 | | 4.7% | 0.424 | | 6.5% |
| | Mean | 1.040 | 1.012 | 2.8% | 0.407 | 0.398 | 2.3% | 1.045 | 1.012 | 3.3% | 0.419 | 0.398 | 5.3% |
| | Standard Deviation | 0.027 | | 2.7% | 0.010 | | 2.5% | 0.015 | | 1.5% | 0.006 | | 1.6% |
| SYS4 | Count | 24 | | 24 | 24 | | 24 | 3 | | 3 | 3 | | 3 |
| | Minimum | 0.748 | | 0.0% | 0.262 | | -1.5% | 0.765 | | 2.3% | 0.278 | | 4.5% |
| | Maximum | 0.804 | | 7.5% | 0.289 | | 8.6% | 0.849 | | 13.5% | 0.309 | | 16.3% |
| | Mean | 0.778 | 0.748 | 4.0% | 0.277 | 0.266 | 4.3% | 0.801 | 0.748 | 7.1% | 0.291 | 0.266 | 9.2% |
| | Standard Deviation | 0.014 | | 1.9% | 0.007 | | 2.6% | 0.043 | | 5.8% | 0.017 | | 6.3% |
| SYS5 | Count | 25 | | 25 | 25 | | 25 | 3 | | 3 | 3 | | 3 |
| | Minimum | 0.964 | | 3.3% | 0.356 | | 0.6% | 0.923 | | -1.1% | 0.355 | | 0.4% |
| | Maximum | 1.030 | | 10.4% | 0.388 | | 9.6% | 1.013 | | 8.6% | 0.396 | | 11.8% |
| | Mean | 0.982 | 0.933 | 5.3% | 0.368 | 0.354 | 4.1% | 0.957 | 0.933 | 2.5% | 0.369 | 0.354 | 4.3% |
| | Standard Deviation | 0.016 | | 1.8% | 0.009 | | 2.5% | 0.049 | | 5.3% | 0.023 | | 6.5% |
| | | | | | | | | | | | | | |

| Standard Number | | Ultratrace results | | | | | | Genalysis results | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% | Relative difference (Co) | Ni% | Expected Ni% | Relative difference (Ni) | Co% | Expected Co% | Relative difference (Co) |
| GBM302-8 | Count | 22 | | 22 | 22 | | 22 | 19 | | 19 | 19 | | 19 |
| | Minimum | 1.07 | | -0.7% | 0.05 | | 3.5% | 1.069 | | -0.8% | 0.050 | | 2.7% |
| | Maximum | 1.200 | | 11.4% | 0.055 | | 13.9% | 1.414 | | 31.2% | 0.065 | | 34.0% |
| | Mean | 1.167 | 1.0775 | 8.3% | 0.053 | 0.048 | 9.4% | 1.207 | 1.0775 | 12.0% | 0.055 | 0.048 | 14.9% |
| | Standard Deviation | 0.032 | | 3.0% | 0.001 | | 3.0% | 0.105 | | 9.7% | 0.005 | | 9.8% |
| GBM900-9 | Count | 17 | | 17 | 17 | | 17 | 14 | | 14 | 14 | | 14 |
| | Minimum | 1.17 | | 0.7% | 0.061 | | 7.6% | 1.099 | | -5.4% | 0.058 | | 1.4% |
| | Maximum | 1.320 | | 13.6% | 0.069 | | 21.7% | 1.549 | | 33.4% | 0.078 | | 38.3% |
| | Mean | 1.249 | 1.162 | 7.5% | 0.064 | 0.057 | 13.6% | 1.275 | 1.162 | 9.8% | 0.067 | 0.057 | 17.5% |
| | Standard Deviation | 0.049 | | 4.2% | 0.002 | | 3.5% | 0.133 | | 11.4% | 0.006 | | 11.2% |
| GBM901-1 | Count | 21 | | 21 | 21 | | 21 | 16 | | 16 | 16 | | 16 |
| | Minimum | 0.825 | | 2.7% | 0.137 | | 1.8% | 0.779 | | -3.0% | 0.134 | | -0.3% |
| | Maximum | 0.932 | | 16.0% | 0.153 | | 13.7% | 1.024 | | 27.4% | 0.174 | | 29.5% |
| | Mean | 0.880 | 0.804 | 9.5% | 0.146 | 0.135 | 8.5% | 0.876 | 0.804 | 9.0% | 0.150 | 0.135 | 11.3% |
| | Standard Deviation | 0.024 | | 3.0% | 0.004 | | 3.1% | 0.077 | | 9.6% | 0.013 | | 9.4% |
| GBM901-2C | Count | 22 | | 22 | 22 | | 22 | 18 | | 18 | 18 | | 18 |
| | Minimum | 0.856 | | -3.1% | 0.031 | | -1.3% | 0.795 | | -10.0% | 0.029 | | -7.0% |
| | Maximum | 0.973 | | 10.2% | 0.036 | | 14.6% | 1.046 | | 18.4% | 0.039 | | 23.9% |
| | Mean | 0.911 | 0.883 | 3.2% | 0.034 | 0.031 | 8.1% | 0.865 | 0.883 | -2.0% | 0.032 | 0.031 | 2.0% |
| | Standard Deviation | 0.033 | | 3.7% | 0.001 | | 4.2% | 0.079 | | 9.0% | 0.003 | | 9.3% |
| GBM902-2 | Count | 9 | | 9 | 9 | | 9 | 11 | | 11 | 11 | | 11 |
| | Minimum | 0.317 | | 5.2% | 0.102 | | 4.6% | 0.289 | | -4.1% | 0.096 | | -1.3% |
| | Maximum | 0.346 | | 14.8% | 0.109 | | 11.8% | 0.368 | | 22.2% | 0.121 | | 24.0% |
| | Mean | 0.333 | 0.301 | 10.6% | 0.106 | 0.098 | 9.1% | 0.328 | 0.301 | 8.9% | 0.108 | 0.098 | 11.1% |
| | Standard Deviation | 0.010 | | 3.3% | 0.002 | | 2.5% | 0.033 | | 10.8% | 0.010 | | 10.4% |

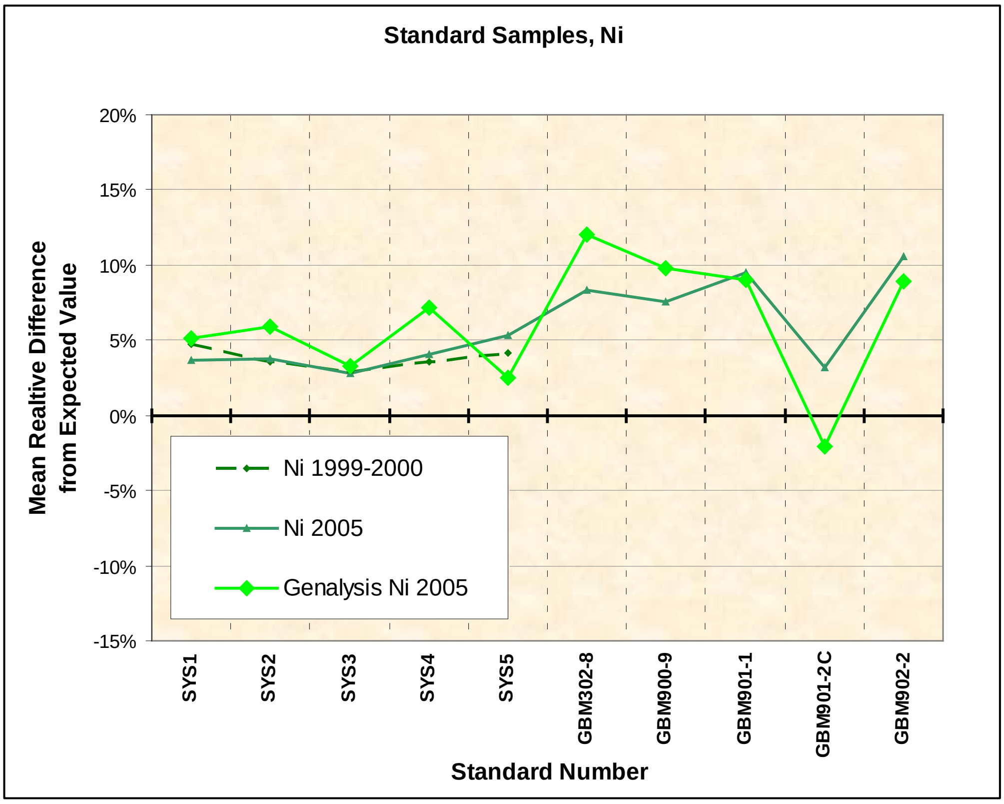



**Figure 11-10: Standards, mean relative differences from expected values (Ni)**

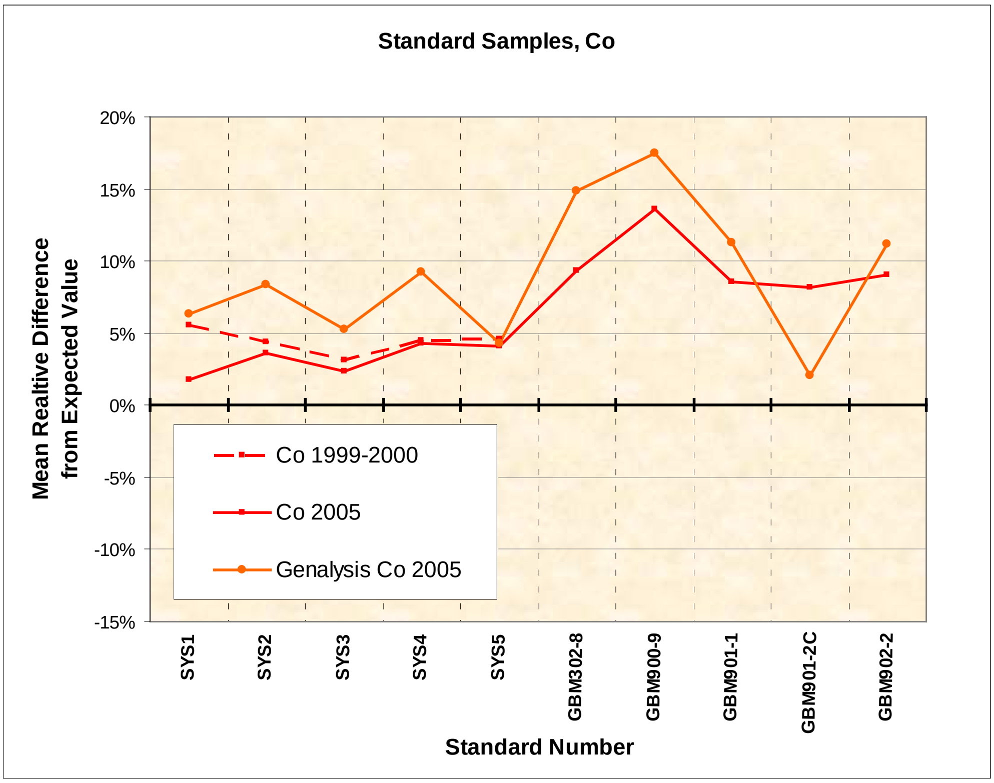



**Figure 11-11: Standards, mean relative differences from expected values (Co)**

The significant apparent bias indicated by the commercial standards was of concern, particularly since these were the 'blind' standards, however, it was at odds with the consistently good results obtained from check assaying. A number of additional checks were then conducted to investigate this further.

Ultratrace re-assayed a batch of 25 standard sample pulps by both the normal method and by fusion XRF. The results compared satisfactorily with the original values and between the two analytical methods. The XRF results did not indicate that there was any significant bias in the original acid digest assays (Table 11-12).

Ultratrace oven dried six commercial standard pulps, left them exposed to ambient laboratory conditions and weighed them at hourly intervals to check for possible moisture absorbing characteristics that might have resulted in low results from laboratories that failed to dry pulps before assaying them. Moisture absorption after 16 hours was consistently negligible, with a maximum of less than 0.3% by weight. This was clearly not a factor.

**Table 11-12: Standard samples – 2005 – Ultratrace re-assays – summary statistics**

| | Assays | | | Relative Differences | | |
|---|---|---|---|---|---|---|
| | Original ICP | Repeat ICP | XRF | Rpt: Orig | XRF: Orig | XRF: Rpt |
| Cobalt | | | | | | |
| Count | 25 | 25 | 25 | 25 | 25 | 25 |
| Minimum | 0.03 | 0.033 | 0.035 | -6.5% | -6.1% | -4.9% |
| Maximum | 0.404 | 0.399 | 0.406 | 4.8% | 7.7% | 3.8% |
| Mean | 0.156 | 0.151 | 0.153 | -1.3% | -1.0% | 0.3% |
| Standard Deviation | 0.11 | 0.11 | 0.11 | 2.5% | 2.7% | 1.9% |
| Nickel | | | | | | |
| Count | 25 | 25 | 25 | 25 | 25 | 25 |
| Minimum | 0.307 | 0.309 | 0.326 | -8.1% | -9.0% | -3.5% |
| Maximum | 1.29 | 1.22 | 1.17 | 5.7% | 5.5% | 2.8% |
| Mean | 0.915 | 0.901 | 0.883 | -0.8% | -1.7% | -0.8% |
| Standard Deviation | 0.25 | 0.25 | 0.23 | 2.6% | 2.6% | 1.4% |

The suppliers of the commercial standards (Geostats Pty Ltd) re-examined the data used to establish recommended Nickel and cobalt values for the standards (which included assays from Ultratrace and Genalysis). The suppliers also reviewed the performance since 2000 of Ultratrace and Genalysis on standards included in their large, regular round robins. This included comparing these laboratories with available neutron activation analyses for Co by Becquerel Laboratories. They reported as follows:

- While individual Nickel and cobalt assay results from both laboratories were consistently within acceptable limits, the Ultratrace values were almost always either equal to or higher than the recommended values. On average, they were approximately 7% relative higher for Nickel and cobalt than the average of all laboratories (Table 11-13). This was reasonably similar to the average 2005 performance on standards that were included with the Syerston samples.
- On average, Genalysis showed only a slight positive bias.
- When assays were restricted to those from 13 preferred laboratories regarded by Geostats as high quality, the average values for the standards increased only slightly and Ultratrace remained high.

**Table 11-13: Summary results on standards – October 2000 to October 2004 – Geostats Pty Ltd**

| Standard | Expected value | Standard deviation | Average Assay | | | | Relative Standardised Difference | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | Preferred Labs | Genalysis | Ultratrace | Becquerel | Genalysis | Ultratrace | Becquerel |
| **Nickel** | | | | | | | | | |
| GBM302-8 | 10775 | 668 | 10919 | 11300 | 11600 | | 4.9% | 7.7% | |
| GBM900-9 | 11615 | 744 | 11651 | 11400 | 12300 | | -1.9% | 5.9% | |
| GBM901-1 | 8037 | 489 | 8120 | 8610 | 8860 | | 7.1% | 10.2% | |
| GBM901-2 | 8804 | 581 | 9057 | 8650 | 8830 | | -1.8% | 0.3% | |
| GBM902-2 | 3014 | 285 | 3028 | 3190 | 3270 | | 5.8% | 8.5% | |
| Mean | 8449 | 553 | 8555 | 8630 | 8972 | | 2.8% | 6.5% | |
| **Cobalt** | | | | | | | | | |
| GBM302-8 | 483 | 31 | 482 | 507 | 519 | 508 | 5.0% | 7.5% | 5.2% |
| GBM900-9 | 567 | 67 | 591 | 568 | 620 | 605 | 0.2% | 9.4% | 6.7% |
| GBM901-1 | 1346 | 77 | 1343 | 1279 | 1460 | 1370 | -5.0% | 8.5% | 1.8% |
| GBM901-2 | 316 | 36 | 330 | 324 | 324 | 343 | 2.5% | 2.5% | 8.5% |
| GBM902-2 | 975 | 109 | 999 | 951 | 1050 | 1180 | -2.5% | 7.7% | 21.0% |
| Mean | 737 | 64 | 749 | 726 | 795 | 801 | 0.0% | 7.1% | 8.6% |

It was clear that, for the commercial standards in particular, Ultratrace has consistently tended to report high relative to the average result from a large group of reputable laboratories. However, this does not necessarily mean that the Ultratrace results have been wrong.

The apparent bias shown by the standards was not at all compatible with the consistently good results from independent assaying of substantial numbers of samples at Genalysis and previously at ALS. These laboratories are all regarded as being of good quality. Ultratrace and Genalysis, in particular, have good reputations for, and are highly experienced in, the analysis of lateritic Ni-Co mineralisation and Ni-PGE mineralisation in general.

The average result from limited neutron activation analysis (Co only) tended to support the Ultratrace values.

While the possibility of a positive bias in Nickel and cobalt assays of the order of 5% - 10% relative cannot be ruled out, McDonald Speijers is aware that both Ultratrace and Genalysis pay a good deal of attention to their acid digest procedures and McDonald Speijers suspects that the apparent bias shown by the standards is likely to reflect more effective digestion of samples than is achieved by most other laboratories.

## 2015

A single, commercial standard sample, identified as OREAS 45e, was submitted at a rate of about 1 in 20 samples. It was supplied by Ore Research and Exploration Pty Ltd of Bayswater, Victoria, and was intended primarily for use as a Sc standard.

While it had certified values for a large range of other elements, the expected Ni value for a 4-acid digest analytical method was only 0.045% (454 ppm), with an expected Co value of only 0.006% (57 ppm). Average resource grades are 15 to 20 times higher, at ~0.65% Ni and 0.105% Co.

McDonald Speijers obtained results compiled by for 136 standard sample submissions during the 2014 - 2015 drilling programs. Results for Nickel and cobalt were satisfactory. However, the expected Nickel and cobalt for this standard were too low to provide any assurance about the accuracy of Nickel and cobalt assays at potential resource grades.

# 11.7 Bulk density

## 11.7.1 Diamond drill core

McDonald Speijers was unable to locate original density data for all the diamond drill holes. The average values reported by Black Range and SLA are shown in Table 11-14.

## 11.7.2 Downhole gamma logging

During the initial review phase, McDonald Speijers located and compiled downhole density probe data for a total of seven diamond drill holes and 135 RC holes.

Values reported at 0.1 m intervals were composited over 1 m intervals. Moisture values from the assay database were then merged and used to estimate dry density values from the wet density values that were reported by Surtron.

The 1 m intervals were flagged by zone and simple statistics were calculated. The length-weighted average values obtained are shown in Table 11-14.

There was little dry density data available for diamond drill holes. Downhole surveys only logged one hole (SDD6) and the holes done by Surtron were either unassayed or no moisture contents were present in the assay database.

## 11.7.3 Calweld holes

McDonald Speijers was only able to locate density data for six of the nine Calweld holes. This was merged with the assay data and flagged with geological zone codes. The average length-weighted values that McDonald Speijers obtained were reasonably similar to those previously reported by Black Range and SLA.

## 11.7.4 Summary

The average dry bulk density values indicated by the various methods are compared in Table 11-14.

**Table 11-14: Comparison of average dry density values**

| Zone | DD core volumetric measurements (5 holes) | DD core water immersion (1 hole) | Gamma/ logging (136 holes with dry densities) | Calweld holes (6 holes) |
|---|---|---|---|---|
| OVB | 1.84 | - | 1.66 | 1.79 |
| TZ | 1.57 | 1.47 | 1.70 | 1.84 |
| GZ | 1.14 | 1.14 | 1.32 | 1.24 |
| SGZ | 1.20 | 1.16 | 1.20 | 1.16 |
| SAP | 1.65 | 1.94 | 1.49 | - |

These comparisons are influenced by some variations in the interpretations of zone boundaries since the diamond drill hole averages were based on earlier SLA interpretations.

Although there is a substantially larger amount of gamma logging data, it is overwhelmingly from RC holes where irregular hole diameters and other technical issues may have adversely affected results. McDonald Speijers is therefore reluctant to accept the data, mainly because of the higher average value that it indicated for the GZ.

## 11.7.5 Factors used for resource estimation

The factors previously used by SLA in the resource model that formed the basis of the 2000 FS are summarised in Table 11-15.

**Table 11-15: Bulk density factors used in resource estimation**

| Zone | Grade range | McDonald Speijers check values from Calweld holes | Factor used |
|---|---|---|---|
| OVB | All | 1.79 | 1.85 |
| TZ | <0.6% NiEq | 1.93 | 1.80 |
| TZl | >0.6% NiEq | 1.65 | 1.55 |
| GZ | All | 1.24 | 1.20 |
| SGZ | All | 1.16 | 1.25 |
| SAP | <0.6% NiEq | No Data | 1.40 |
| SAP | >0.6% NiEq | No Data | 1.30 |

These factors were based more or less entirely on the results from the Calweld holes (except for the saprolite factors which were based on down hole gamma logging results). McDonald Speijers agrees that these were probably the most reliable values and that this was a reasonable choice.

McDonald Speijers also regards that it was reasonable to adopt a higher average value for the SGZ than indicated by the Calweld holes, because they failed to fully penetrate the zone and McDonald Speijers would expect average density to increase in its lowermost parts.

Even though SRK's checks on the Calweld data suggested slightly higher average density values, McDonald Speijers was reluctant to increase bulk density factors because of the negative relationship between grade and density. McDonald Speijers concluded that the factors that had previously been used provided a reasonable basis for resource tonnage estimation, given the data available (considering that the amount of reliable data is quite limited).

Considering the data overall, it is clear that average density factors could be prone to errors of the order of ±5% - 10%. The greatest uncertainty is in the SAP, but this contains only a very small proportion of the resource (<2% of tonnage).

## 11.8 2017 Mineral Resource estimate QA/QC drill hole series validation

To provide assay confidence for the 2017 Mineral Resource Estimate, several QAQC validation activities where undertaken on the drill hole assay database. These exercises were carried out separately on the various drilling campaigns to ascertain the assay confidence for each drill hole data set.

The operators on the project carried out various levels of assay quality control, which principally involved inclusion of field duplicates and standards within each batch submitted to the routine assay laboratory. Further verification of the accuracy of the assaying was carried out for several drilling campaigns by submitting laboratory pulps to a check assay laboratory.

The laboratory replicates were investigated for each drilling campaign and reported by laboratory batches to assess the laboratory repeatability. Blank samples to test for laboratory contamination were only submitted during the 2014-2016 drilling programs.

### 11.8.1 Standards

#### SRC001-SRC340 Drilling Program (August 1997 – August 1998)

Initial RC drilling program included systematic insertion of standards using the Gannet SYS1 to SYS5 standards. Black Range Minerals NL (BRM) submitted these standards under alias Standard Numbers e.g. S1 ≡ S4, S8, S12 and S20.

Shewhart assay quality control charts were generated for each Gannet Standard and analyte, grouped by Batch No. A linear regression line was plotted on the chart to indicate any discernible trend between successive batches.

Standard values in SYS5 were recorded as slightly higher than but within +2 Standard Deviations (SDs) with no observed trend in the regression line. The other Gannet Standards were evaluated with comparable results, although the trend lines where less consistent in these standards, often with rising trends observed.

There were some instances when the values exceeded +3 SDs and theses batches should have been referred to the routine laboratory for rectification.

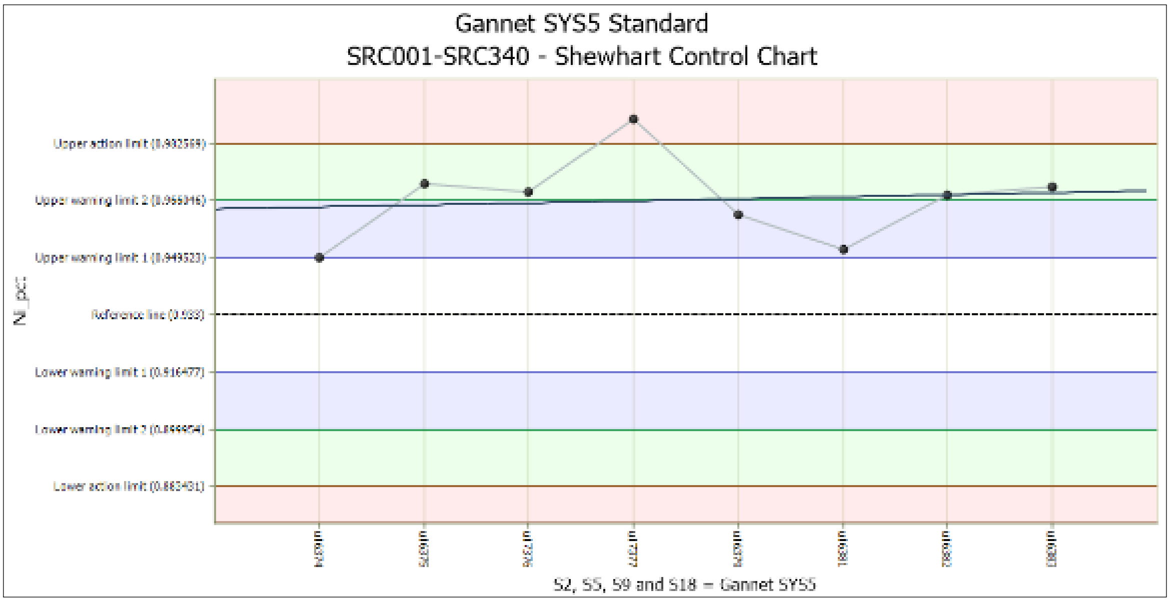



**Figure 11-12: Nickel % Shewhart quality control chart – Gannet SYS5 standard**

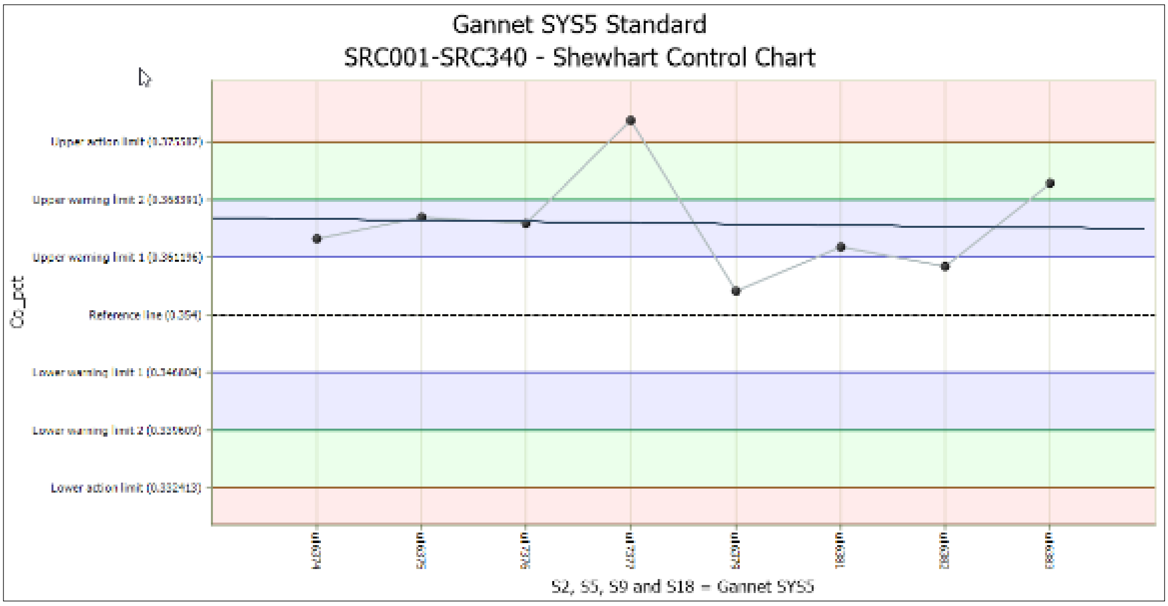



**Figure 11-13: Cobalt % Shewhart quality control chart – Gannet SYS5 standard**

## SRC0341-SRC1076 Drilling Program (August 1998 – August 2000)

This RC drilling program also utilised the BRM Gannet SYS1 to SYS5 standards to monitor the assay quality control. Shewhart Quality Control charts were generated for each standard and analyte.

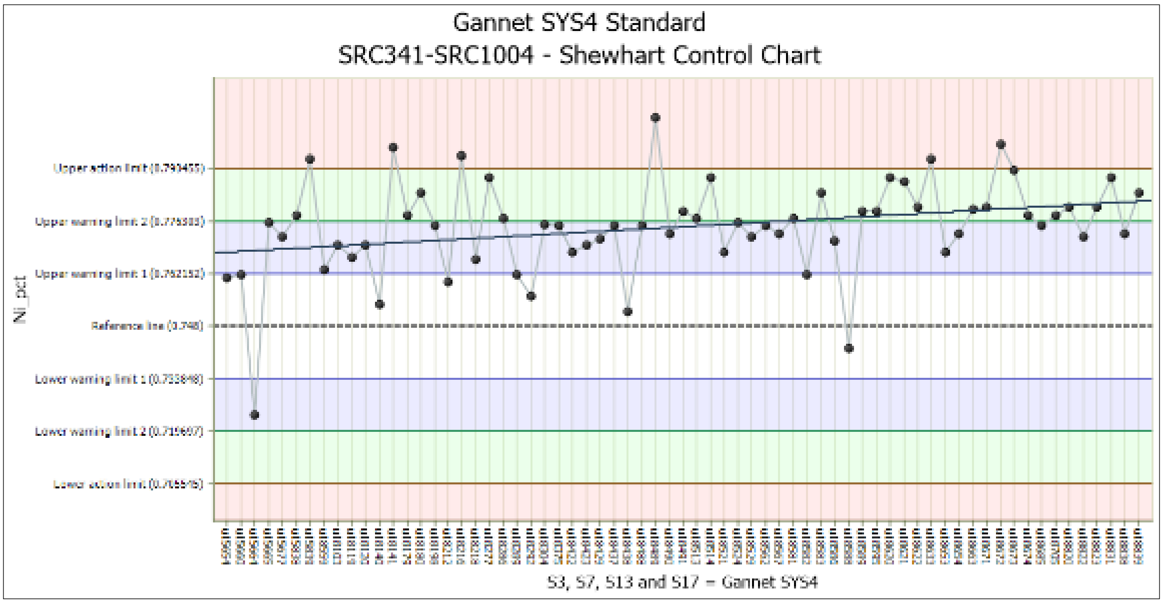



**Figure 11-14: Nickel % Shewhart quality control chart – Gannet SYS4 standard**

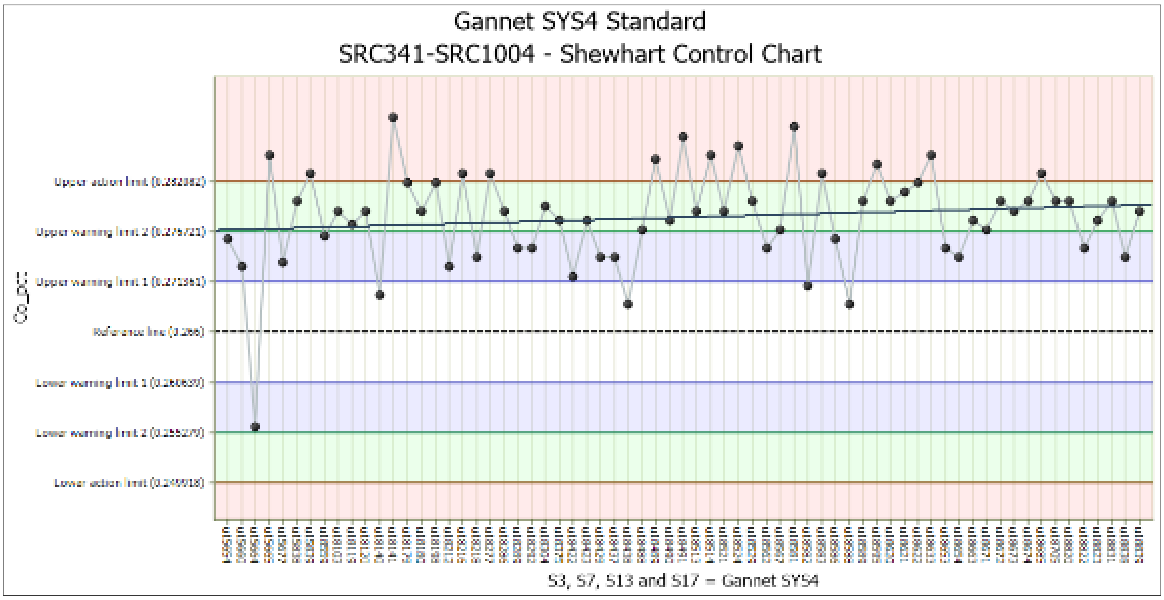



**Figure 11-15: Cobalt % Shewhart quality control chart – Gannet SYS4 standard**

The standard values were reported higher for this drilling program and the trend analysis indicated that the values were slightly enhanced towards the end of the drilling campaign.

There is evidence that several batches have values exceeding +3SDs (above 'upper warning limit'). These batches should have been referred back to the laboratory for rectification.

## SRC1077-SRC1251 Drilling Program (February 2005 – March 2005)

This drilling program was carried out by Ivanplats/Syerston who continued to utilise the BRM Gannet Standards.

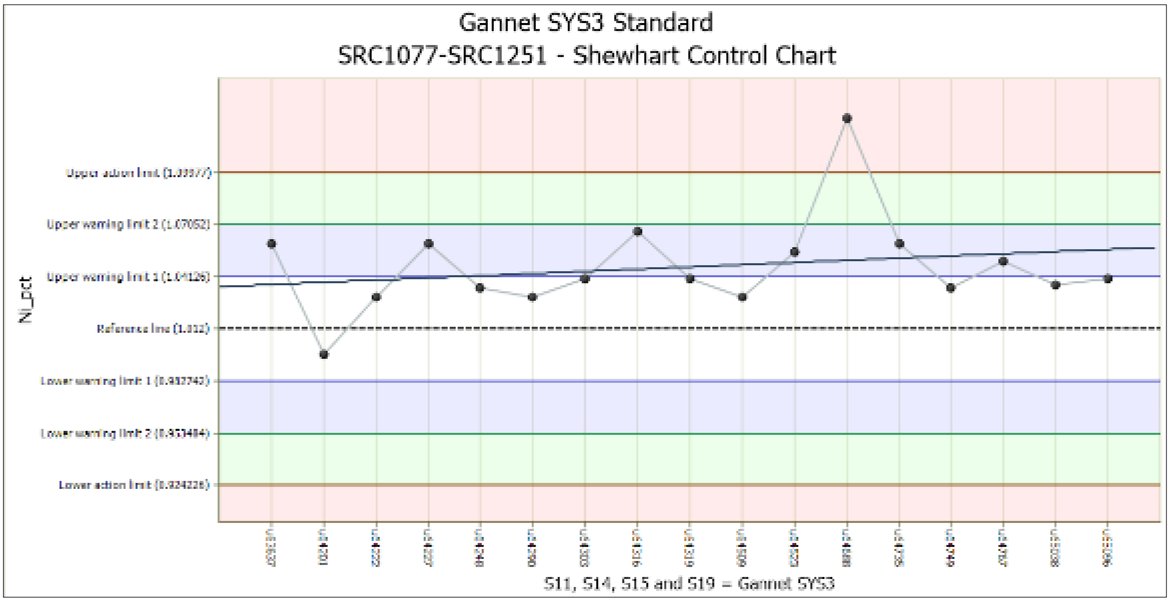



**Figure 11-16: Nickel % Shewhart quality control chart – Gannet SYS3 standard**

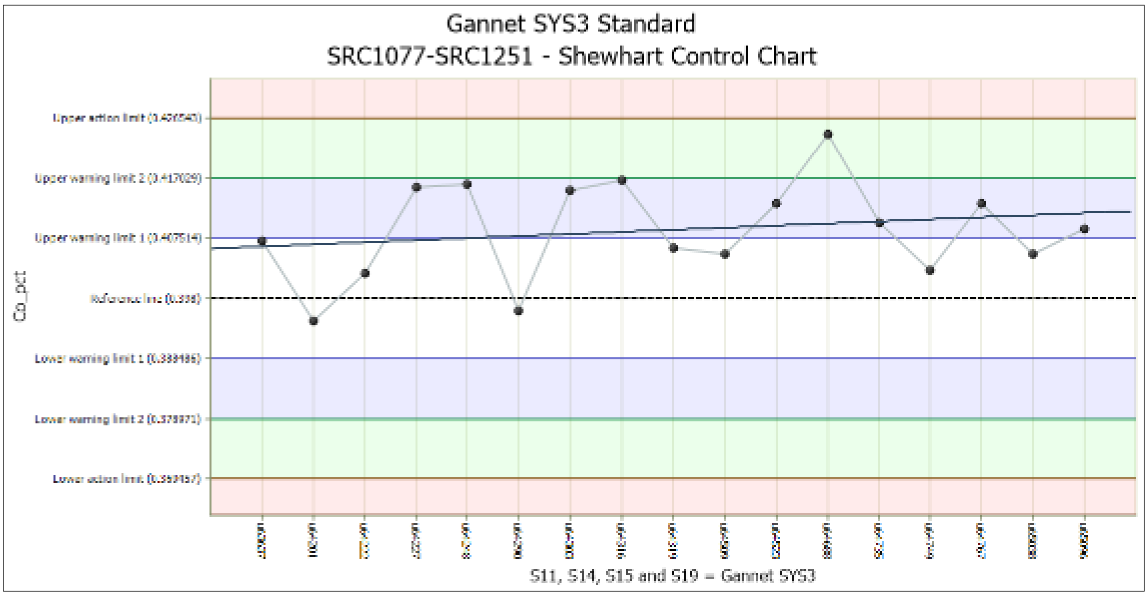



**Figure 11-17: Cobalt % Shewhart quality control chart – Gannet SYS3 standard**

The standard data was treated identically to the two previous drill programs. The assay quality control charts indicate an improvement in the laboratory quality control where all values are slightly higher but within 2SDs and the number of 'outliers' (+3SDs) in very minor. The batches are statistically reliable.

## SRC1263-SRC1276 Drilling Program (August 2014)

This RC drilling program was carried out by Ivanplats/Syerston who introduced a new standard prepared by Ore Research and Exploration Pty Ltd and the Certified Reference Material (CRM) was referred to as Standard OREAS 45e.

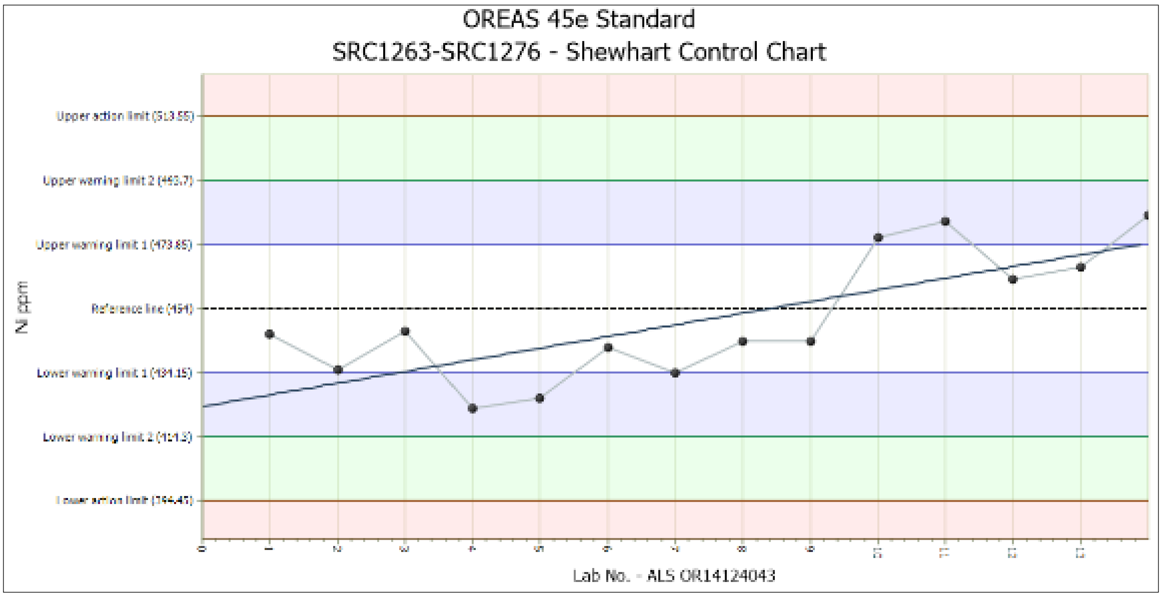



**Figure 11-18: Nickel % Shewhart quality control chart – OREAS 45e standard**

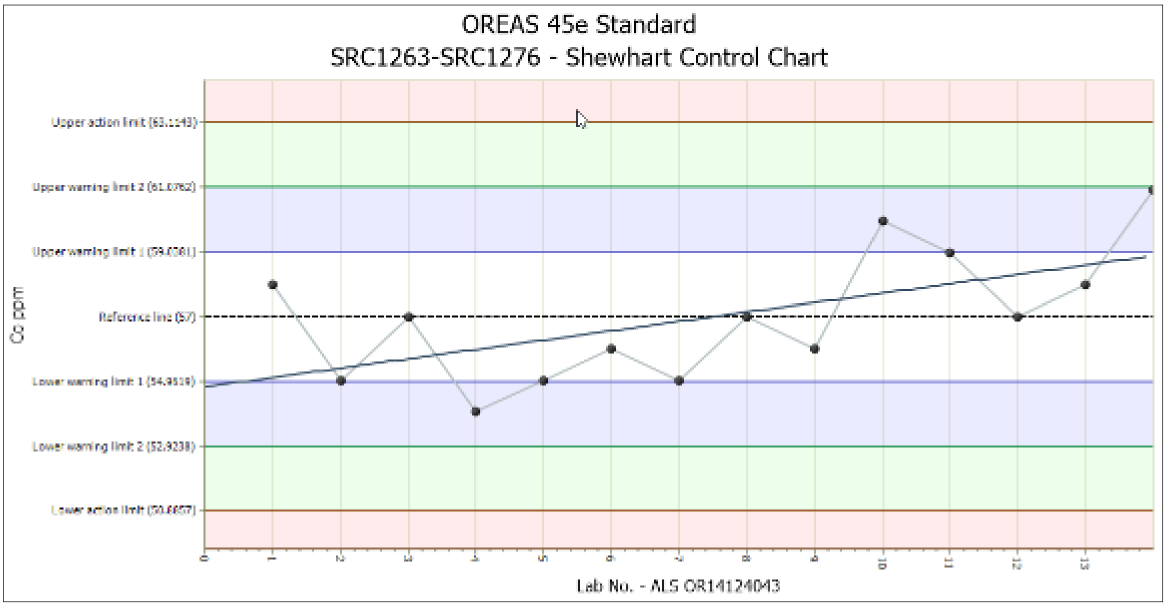



**Figure 11-19: Cobalt % Shewhart quality control chart – OREAS 45e standard**

The standard results were all within the one laboratory Job No. and although the values for both Ni and Co did not exceed ±2SDs, there is a very discernible trend from the first to last result and it relates both to the Ni and Co values. This suggest possible drift in the instrument and warranted checking with the laboratory.

## SRC1277-SRC1310 Drilling Program (April 2015)

The SRC1277-SRC 1310 drilling program was carried out by Scandium21 and the OREAS 45e CRM/Standard was used to monitor the assay quality control for this program.

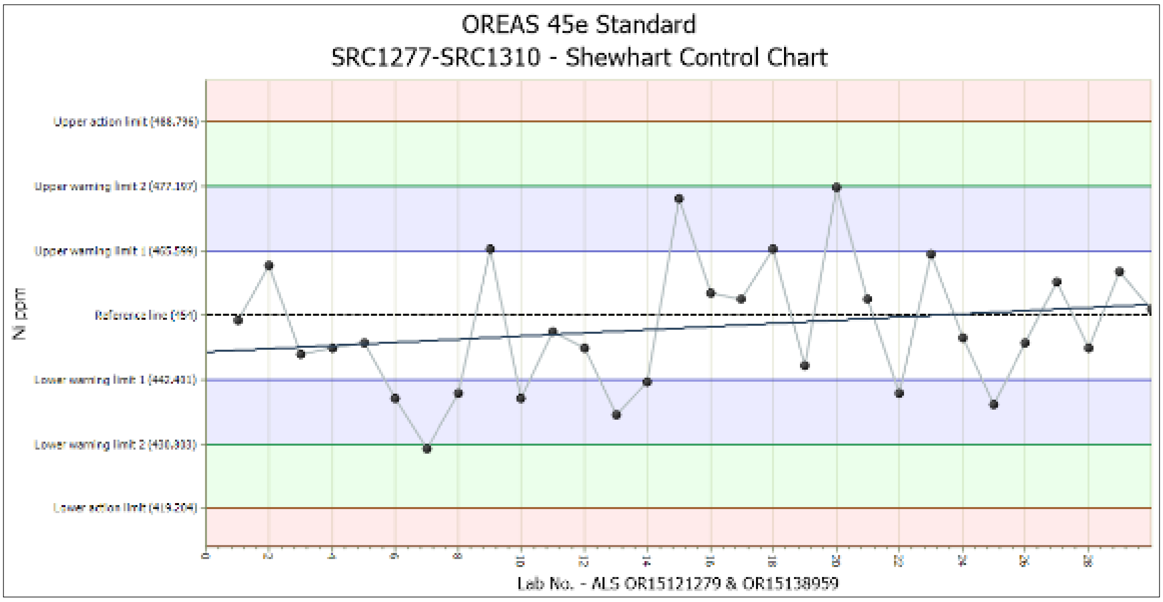



**Figure 11-20: Nickel ppm Shewhart quality control chart – OREAS 45e standard**

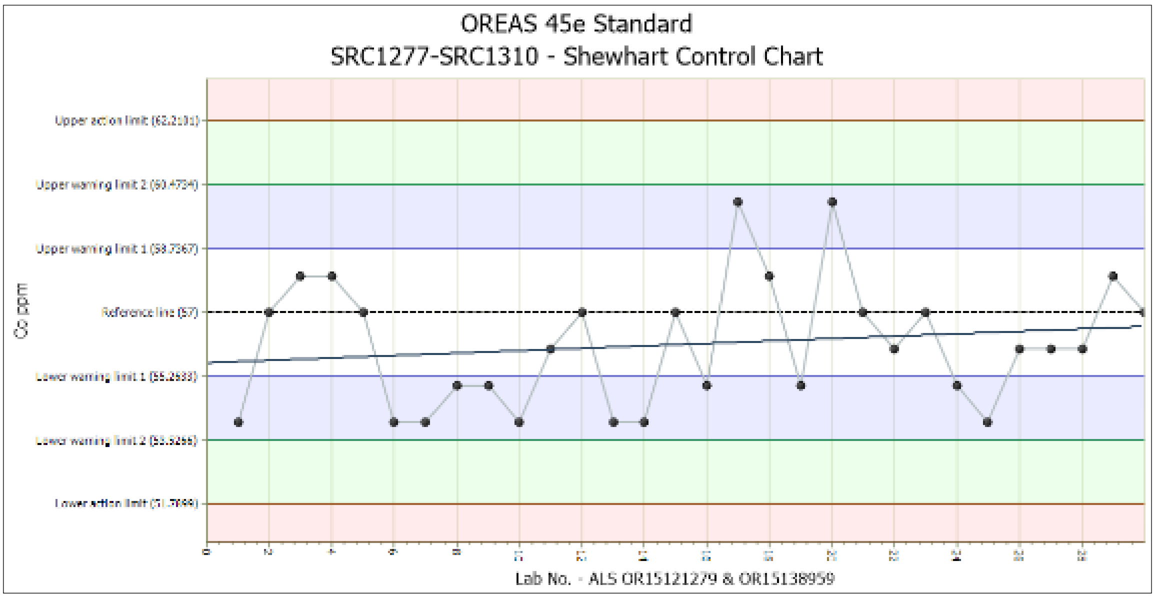



**Figure 11-21: Cobalt ppm Shewhart quality control chart – OREAS 45e standard**

The results for Ni improved with the values distributed both sides of the reference line and within ±2SDs and only a minor trend up in the sequence of values. The Co values lie primarily below the reference line with no values exceeding ±2SDs and a minor up trend in the sequence of values. The lower reported value for Co may have affected the outcome of the routine sample values but maybe not be enough to warrant the re-assay of the batches.

As this drilling program was primarily aimed at the evaluation of the scandium resource, the Sc assay quality control chart is included, which indicates that the values are position on the -1SD line and display an even trend. The slightly lower standard values should not influence the outcome of the routine assay values.

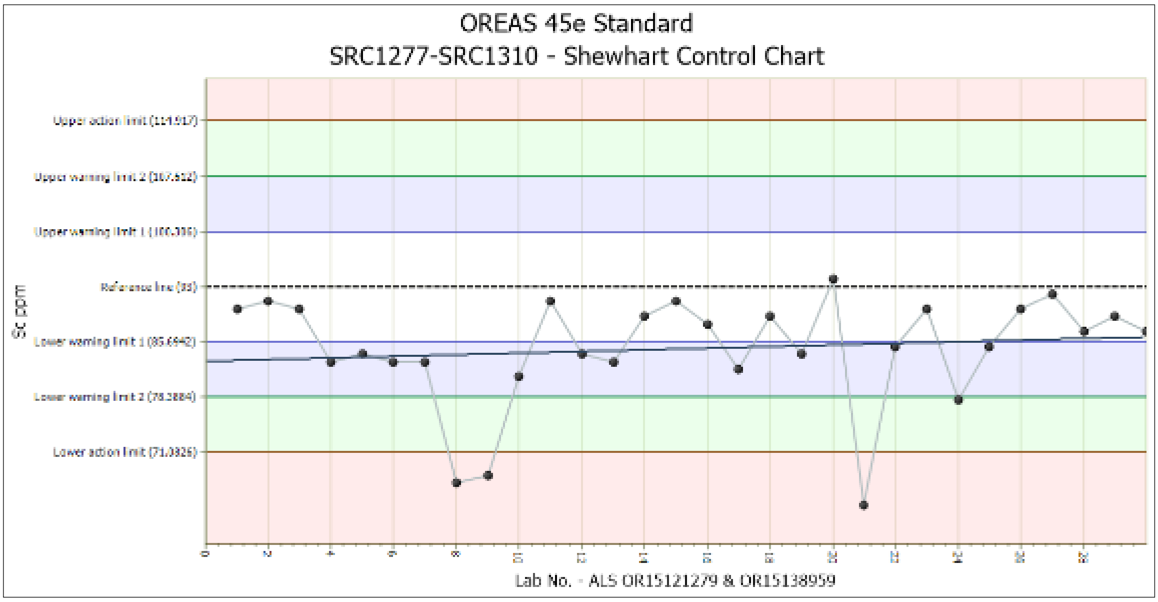



**Figure 11-22: Scandium ppm Shewhart quality control chart – OREAS 45e standard**

## SRC1311-SRC1368 Drilling Program (Nov2015)

This RC drilling program was carried out by Scandium21 and focused on the delineation of the Sc resource. It utilised the ORES 45e CRM/Standard to monitor the assay quality control.

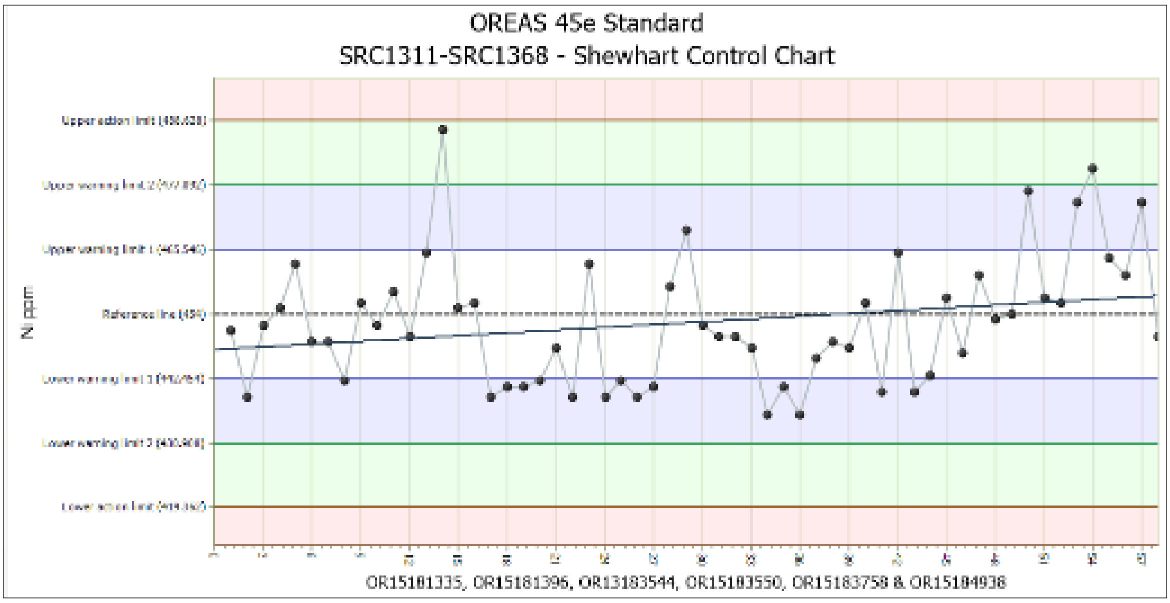



**Figure 11-23: Nickel ppm Shewhart quality control chart – OREAS 45e standard**

The Ni standard values straddle the reference line and the values do not exceed ±2SDs. Two values exceed 2SDs and the batch containing these values should be assessed independently. There is a slight upward trending during the period within which the batches were analysed.

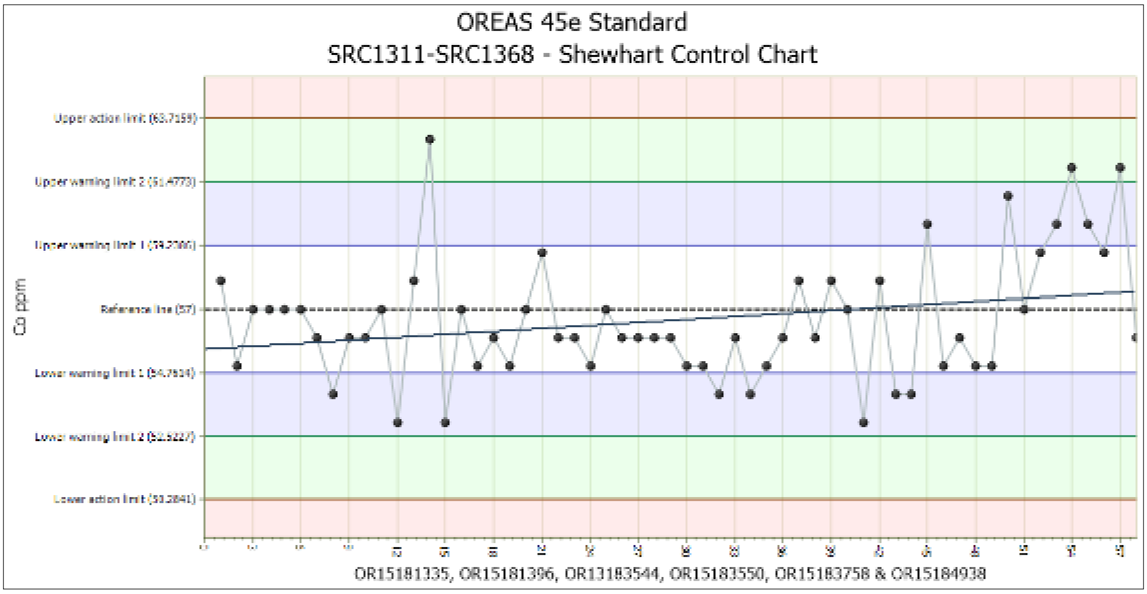



**Figure 11-24: Cobalt ppm Shewhart quality control chart – OREAS 45e standard**

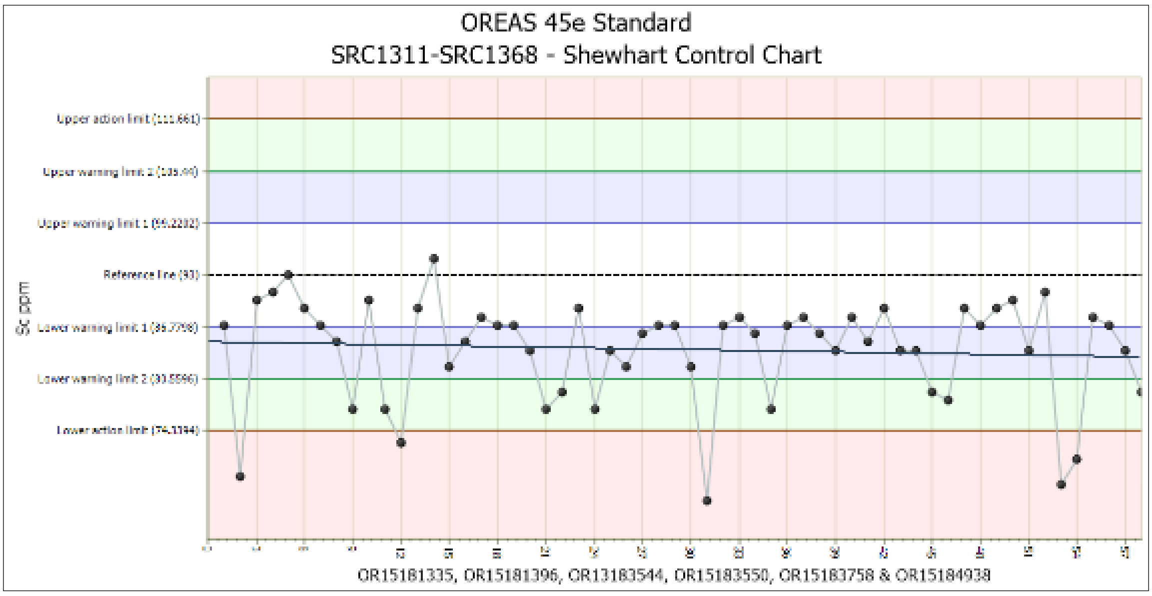



**Figure 11-25: Scandium ppm Shewhart quality control chart – OREAS 45e standard**

The Cobalt values are predominantly with -1SD with a minor number within -2SDs. There is a pronounced upward trend in the last batch resulting in the linear regression line trending up sharply at the end of the chart.

The Scandium values lie either close to or predominantly immediately below the reference lin. For the most part, they lie within -1SD with a minor number -2SDs and a few ‘outliers’.

## 11.8.2 Duplicates

### SRC001-SRC340 Drilling Program (August 1997 – August 1998)

There is no record of duplicate samples being collected and analysed for this drilling program. This allows no opportunity to evaluate the sampling and sample preparation on this program.

## SRC0341-SRC1076 Drilling Program (August 1998 – August 2000)

The duplicate sample values were evaluated against the original assay values using the Correlation Analysis function in the GeoAccess Professional program. A linear regression is undertaken and the Correlation Coefficient is determined. The 95% confidence limits are displayed in the Correlation Graph. A QA/QC Limits test is applied at 10%, 20% and 30%.

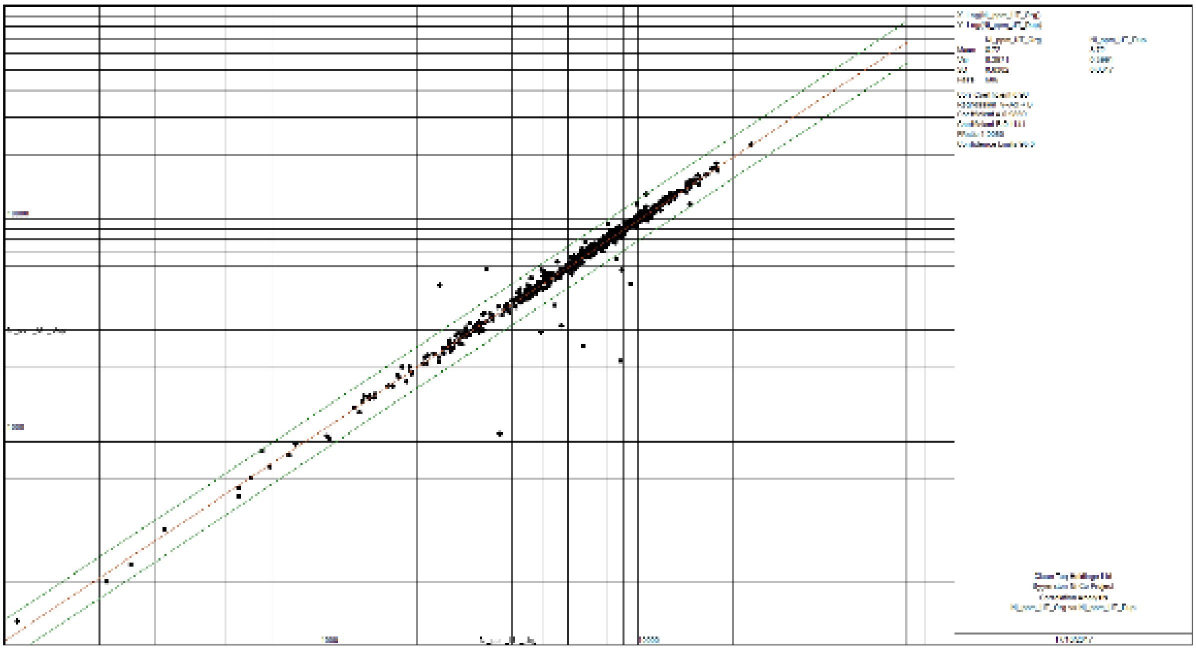


**Figure 11-26: SRC0341 - SRC1076 – Ni ppm originals versus duplicates**

Ni Duplicate Samples:

- Correlation Coefficient: 0.98.
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.5%
- Statistical status: Statistically ‘in-control’.

Co Duplicate Samples:

- Correlation Coefficient: 0.95.
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 2.1%
- Statistical status: Statistically ‘in-control’

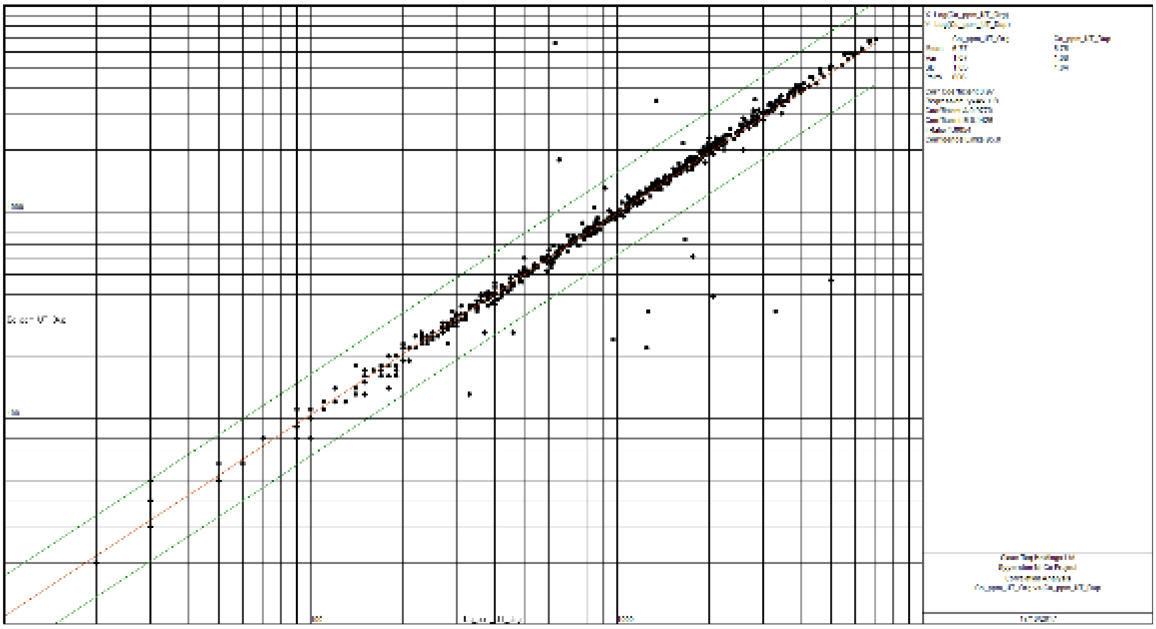


**Figure 11-27: SRC0341 - SRC1076 – Co ppm originals versus duplicates**

## SRC1077-SRC1251 Drilling Program (February 2005 – March 2005)

There is no record of duplicate samples being collected and analysed for this drilling program. This allows no opportunity to evaluate the sampling and sample preparation on this program.

## SRC1263-SRC1276 Drilling Program (Aug2014)

This limited 14 hole drill program represents the initial Sc resource delineation drilling and two duplicate sample where taken in each drill hole.

Original versus duplicate tests included:

- Correlation Coefficient;
- Diagonal line (1:1) versus regression line (bias test); and
- QA/QC Limits Test at 10%, 20% and 30%.

Ni Duplicate Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: Slight bias towards duplicates
- ±10% QA/QC limits: 0%
- Statistical status: Statistically ‘in-control’

Co Duplicate Samples:

- Correlation Coefficient: 0.98
- Diagonal versus regression lines: Minor bias towards duplicates
- ±10% QA/QC limits: 15.4%
- Statistical status: Statistically ‘out-of-control’

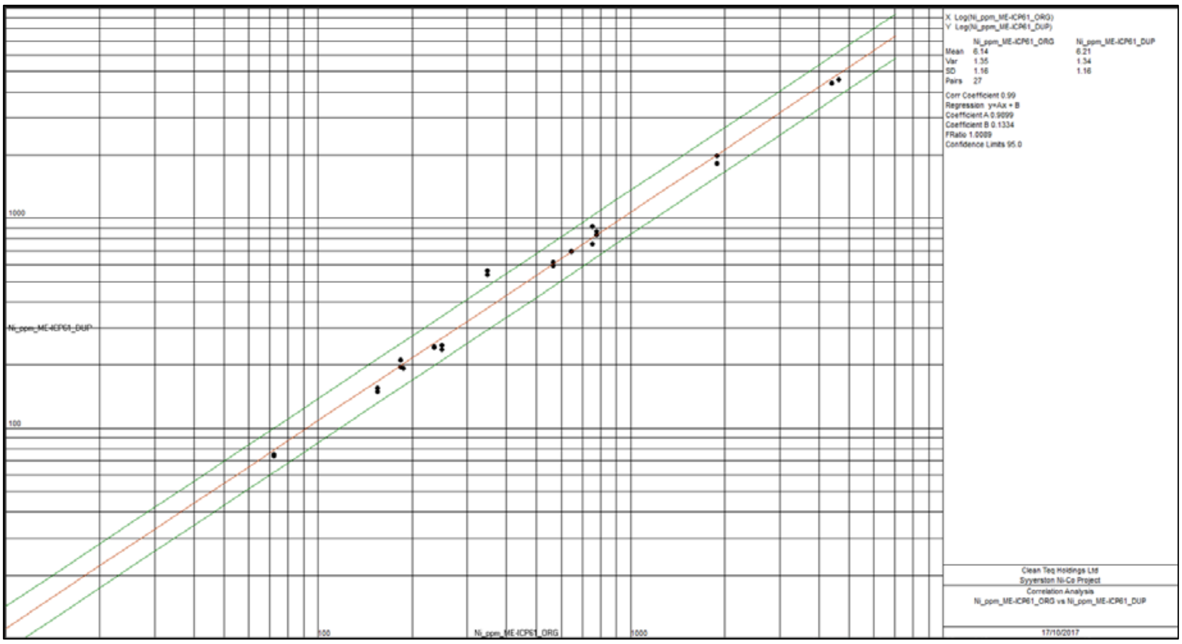



**Figure 11-28: SRC1263 - SRC1276 – Ni ppm originals versus duplicates**

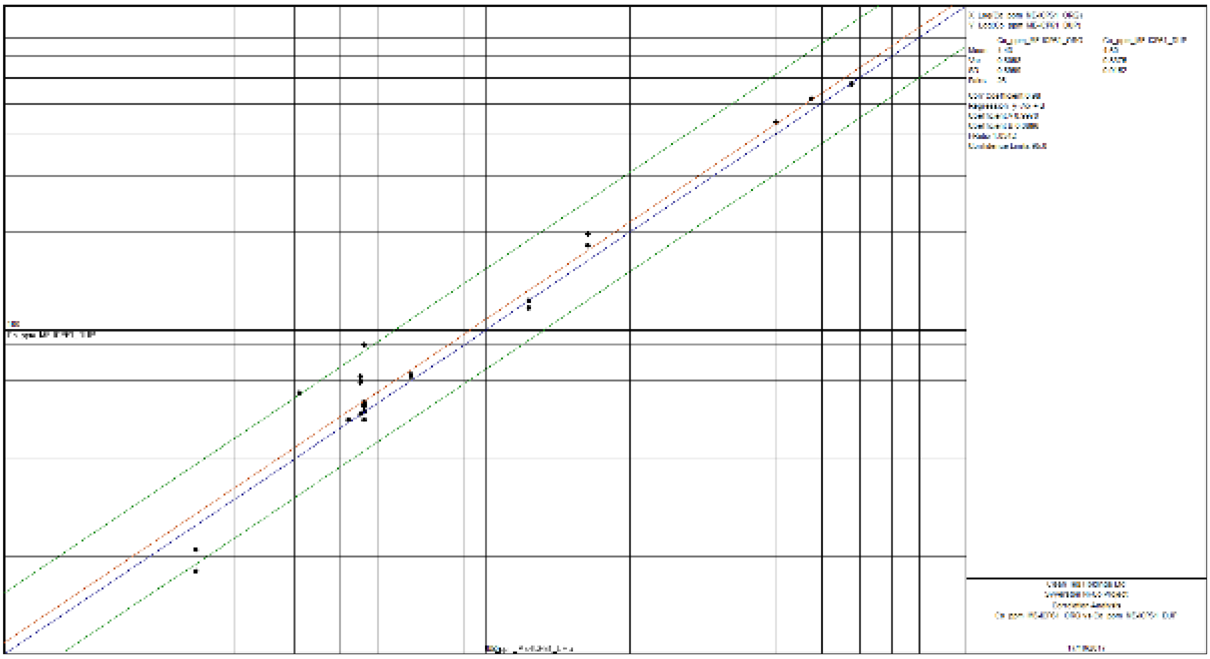


**Figure 11-29: SRC1263 - SRC1276 – Co ppm originals versus duplicates**

## SRC1277-SRC1310 Drilling Program (April 2015)

This 34 hole drilling program was a continuation of the Sc resource delineation drilling and as for the initial phase of drilling, two duplicate samples were taken from each hole. As for the initial program, duplicates were obtained by spearing the field reject sample. The original samples were taken as 2 m composites and later individual 1m samples were collected.

Original versus duplicate tests included:

- Correlation Coefficient
- Diagonal line (1:1) versus regression line (bias test)
- QA/QC Limits Test at 10%, 20% and 30%.

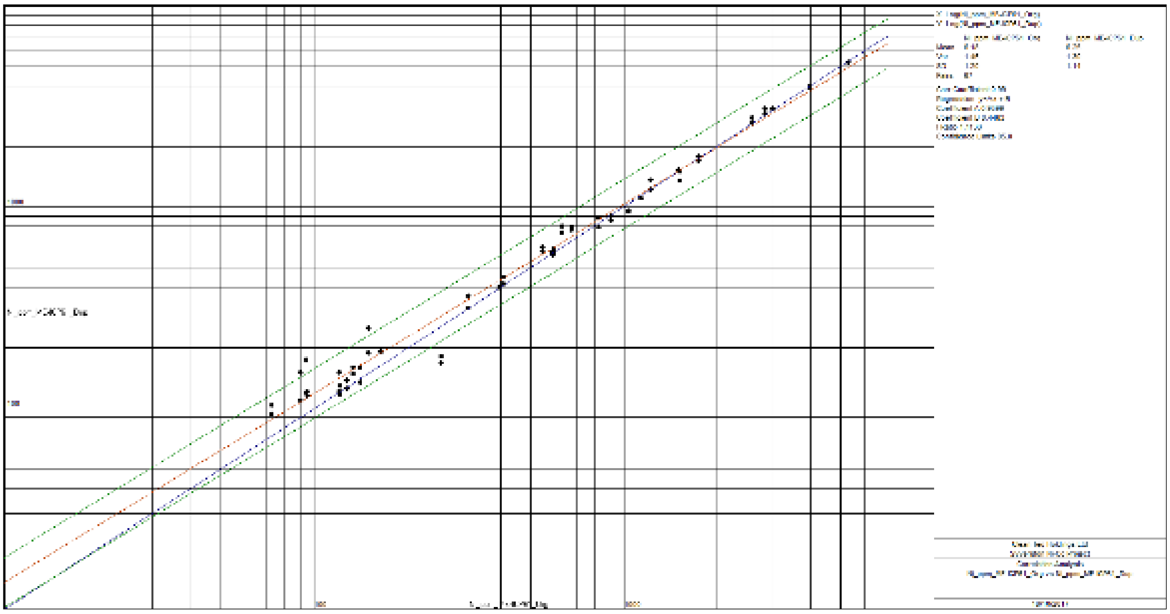


**Figure 11-30: SRC1277 - SRC1310 – Ni ppm 2 m originals versus duplicates**

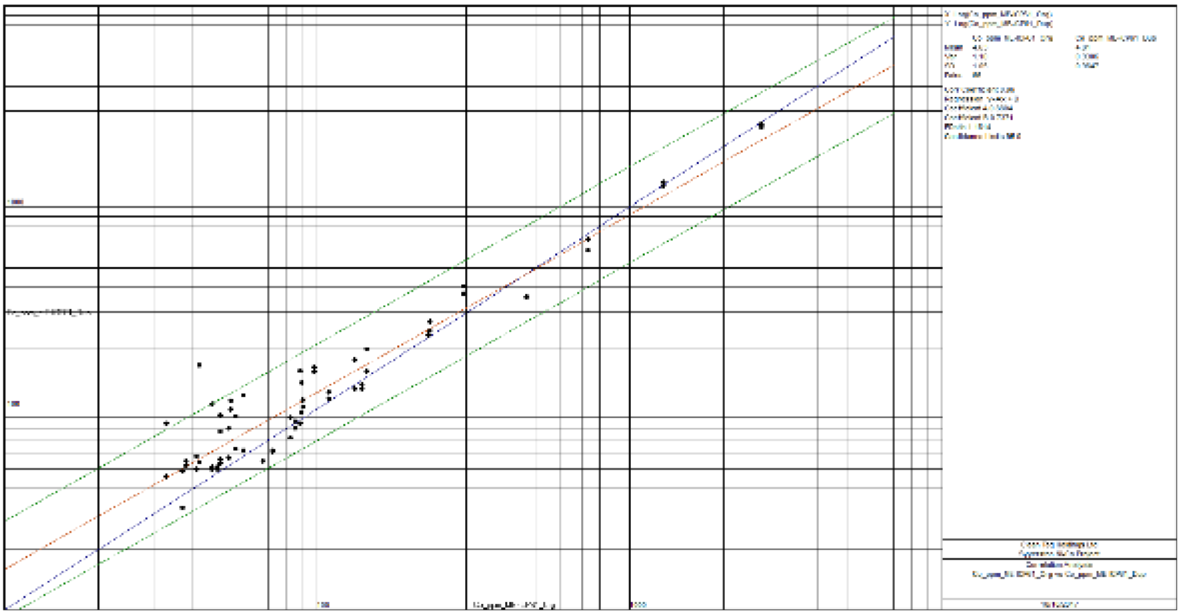


**Figure 11-31: SRC1277 - SRC1310 – Co ppm 2 m originals versus duplicates**

2 m Ni Duplicate Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: Pronounced bias towards duplicates at lower values
- ±10% QA/QC limits: 4.4%
- Statistical status: Statistically ‘in-control’

2 m Co Duplicate Samples:

- Correlation Coefficient: 0.98
- Diagonal versus regression lines: Significant bias towards duplicates at lower values
- ±10% QA/QC limits: 19.7%
- Statistical status: Statistically ‘out-of-control’

2 m Sc Duplicate Samples:

- Correlation Coefficient: 0.93
- Diagonal versus regression lines: Slight bias towards originals across the range of values
- ±10% QA/QC limits: 16.7%
- Statistical status: Statistically 'out-of-control'

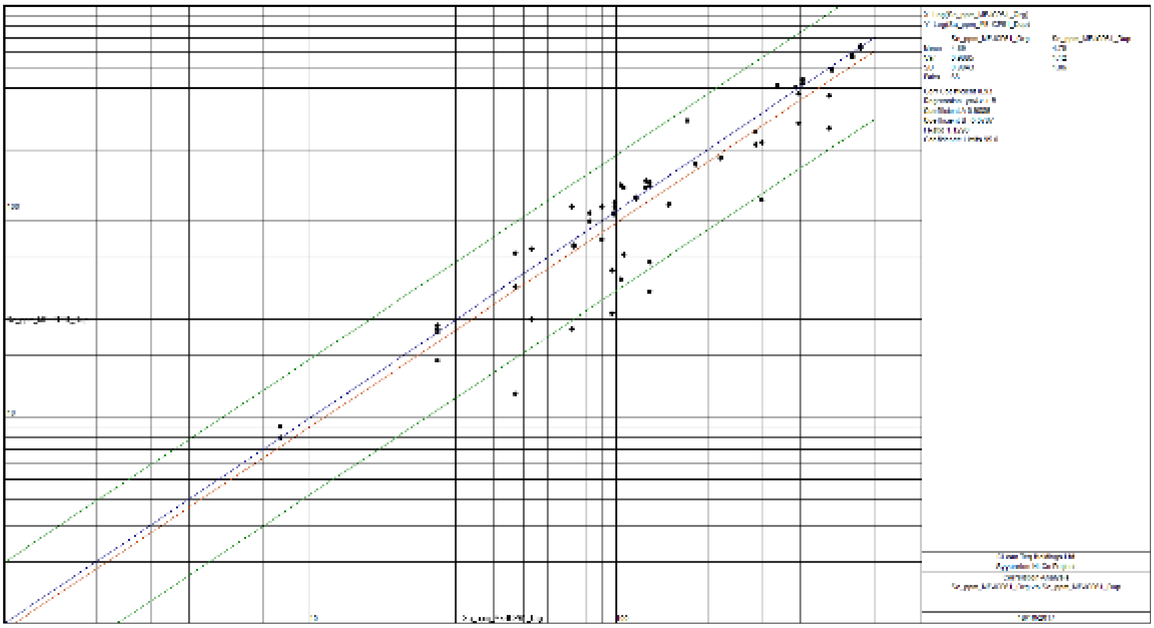


**Figure 11-32: SRC1277 - SRC1310 – Sc ppm 2 m original versus duplicates**

1 m Ni Duplicate Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 10%
- Statistical status: Statistically 'in-control'

1 m Co Duplicate Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0%
- Statistical status: Statistically 'in-control'

1m Sc Duplicate Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0%
- Statistical status: Statistically 'in-control'

**Figure 11-33: SRC1277 - SRC1310 – Ni ppm 1 m originals versus duplicates**

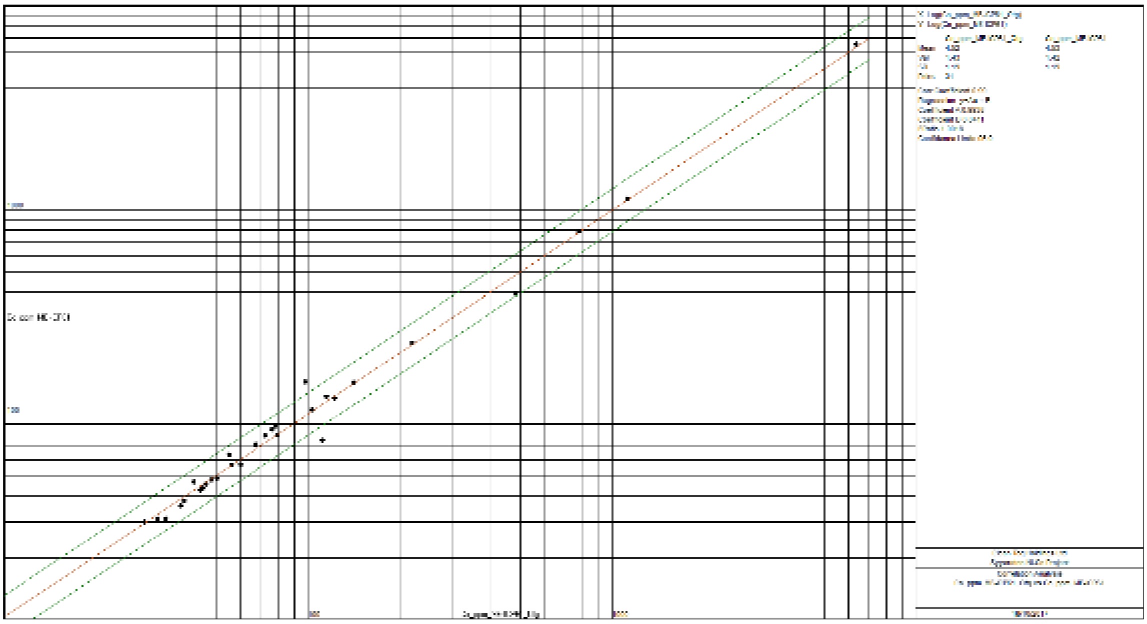


**Figure 11-34: SRC1277 - SRC1310 – Co ppm 1 m originals versus duplicates**

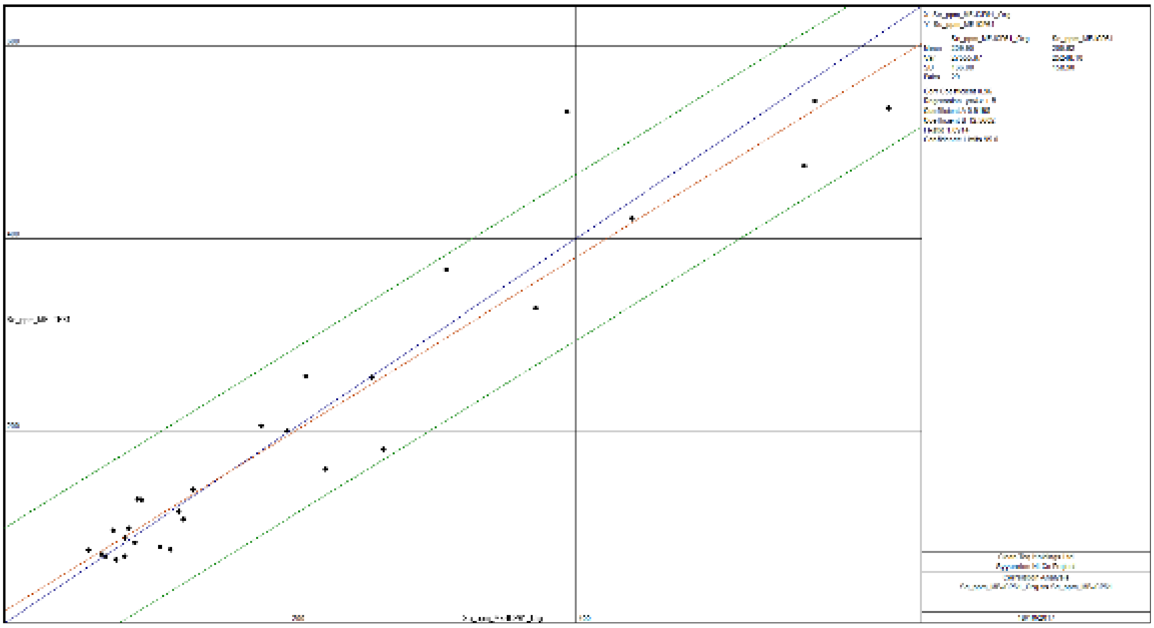


**Figure 11-35: SRC1277 - SRC1310 – Sc ppm 1 m originals versus duplicates**

### 11.8.3 SRC1311-SRC1368 drilling program (November 2015)

The 58 hole drilling program was the most recent drilling focussed on the Sc resource definition. Two duplicate samples were taken from each hole and as for the previous two Sc drilling programs they were obtained by spearing the field reject sample.

Original versus duplicate tests included:

- Correlation Coefficient
- Diagonal line (1:1) versus regression line (bias test)
- QA/QC Limits Test at 10%, 20% and 30%.

<u>SRC1311 – SRC1368 Ni Duplicate Samples:</u>

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: Slight bias to duplicates
- ±10% QA/QC limits: 47.4%
- Statistical status: Statistically 'out-of-control'

<u>SRC1311 – SRC1368 Co Duplicate Samples:</u>

- Correlation Coefficient: 0.97
- Diagonal versus regression lines: Pronounced bias to original samples in higher values
- ±10% QA/QC limits: 74.1%
- Statistical status: Statistically 'out-of-control'

<u>SRC1311 – SRC1368 Sc Duplicate Samples:</u>

- Correlation Coefficient: 0.97
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 51.8%
- Statistical status: Statistically 'out-of-control'

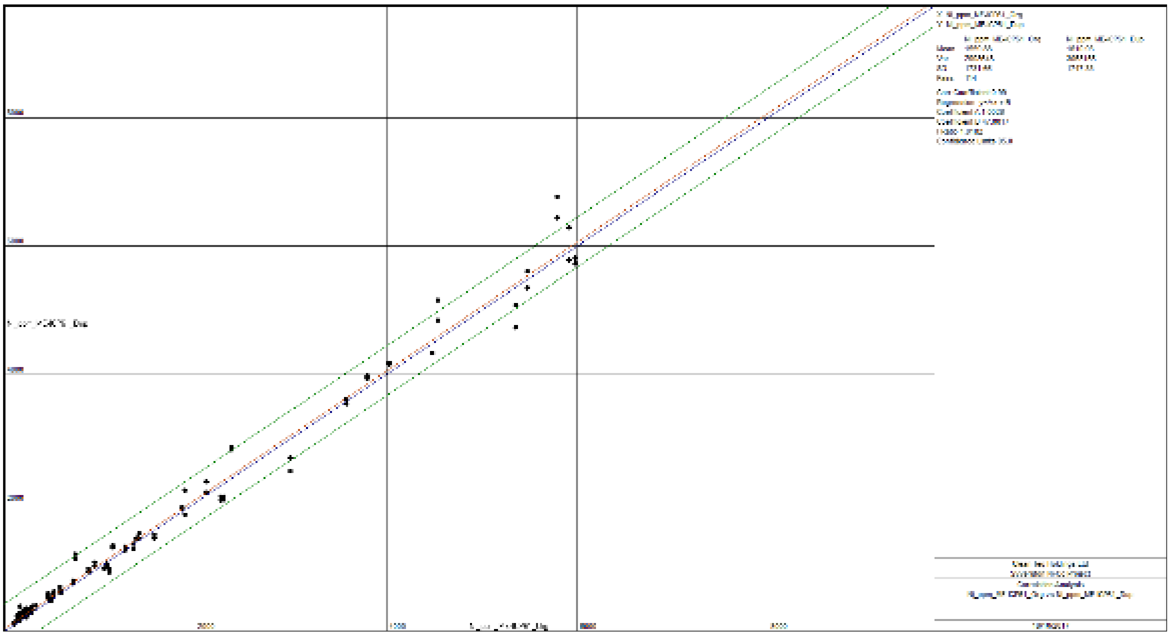


**Figure 11-36: SRC1311 - SRC1368 – Ni ppm originals versus duplicates**

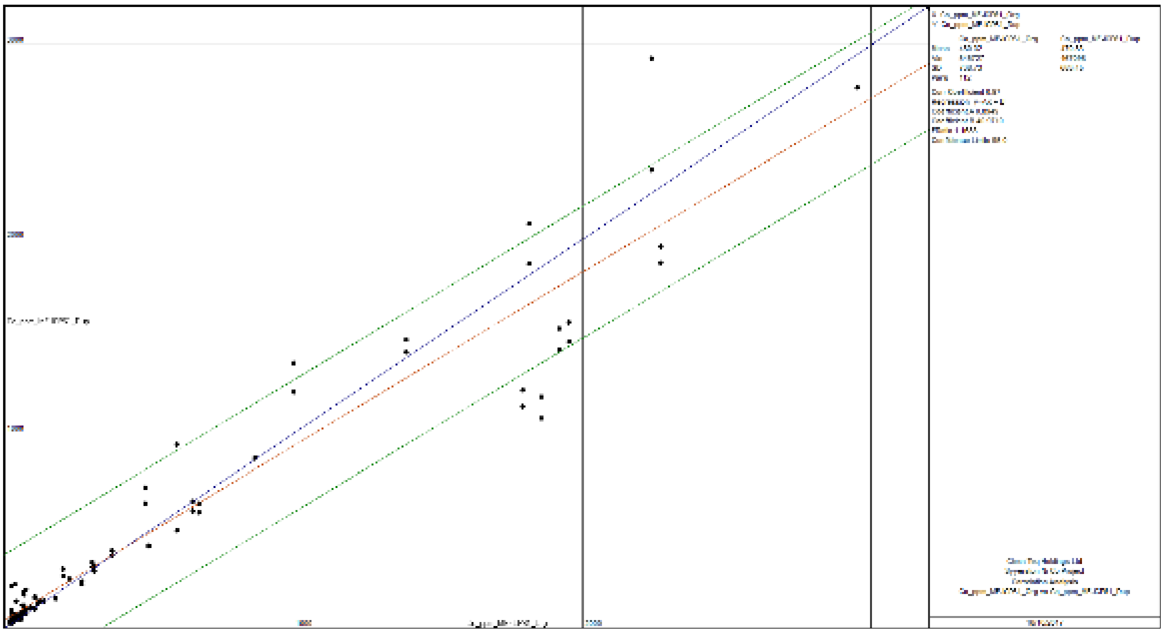


**Figure 11-37: SRC1311 - SRC1368 – Co ppm originals versus duplicates**

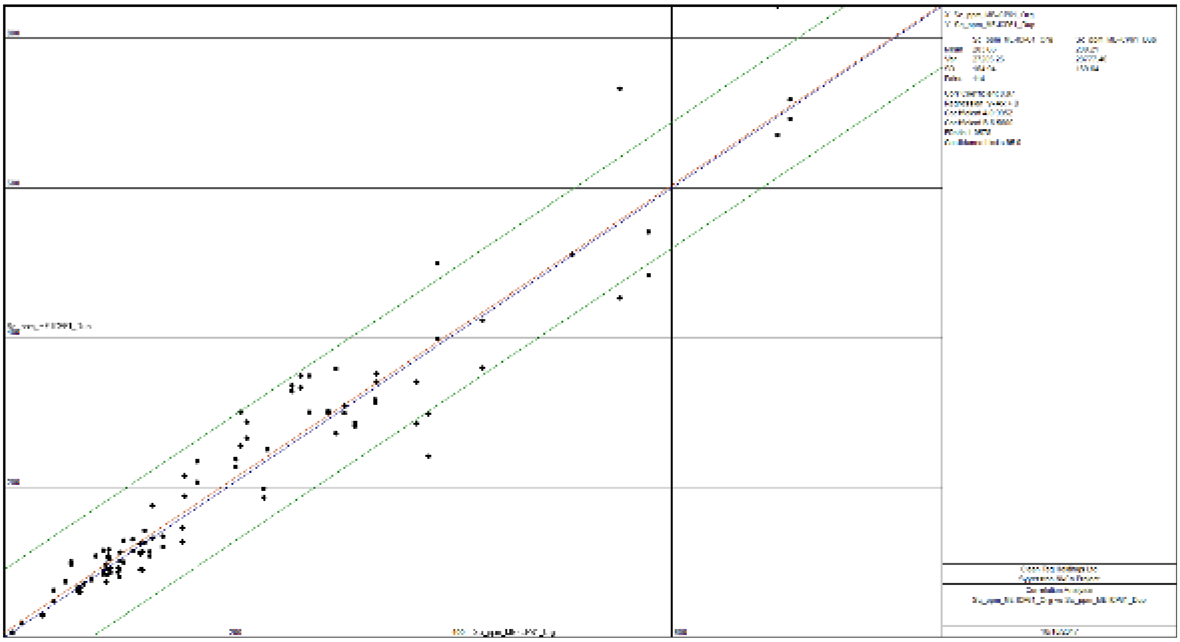


**Figure 11-38: SRC1311 - SRC1368 – Sc ppm originals versus duplicates**

## 11.8.4 Replicates

Laboratory replicate sample values were available for all the major drilling programs and this data was sourced from historic records as Excel workbooks and imported into Micromine for initial analysis.

The replicate data was imported into GeoAccess2013 and several tests to confirm laboratory repeatability were applied.

Original versus duplicate tests included:

- Correlation Coefficient
- Diagonal line (1:1) versus regression line (bias test)
- QA/QC Limits Test at 10%, 20% and 30%.

### SAC120 – SAC267 Drilling Program (August 1995-August 1996)

<u>SAC120 – SAC267 Ni Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 4.3%
- Statistical status: Statistically 'in-control'

<u>SAC120 – SAC267 Co Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.0%
- Statistical status: Statistically 'in-control'

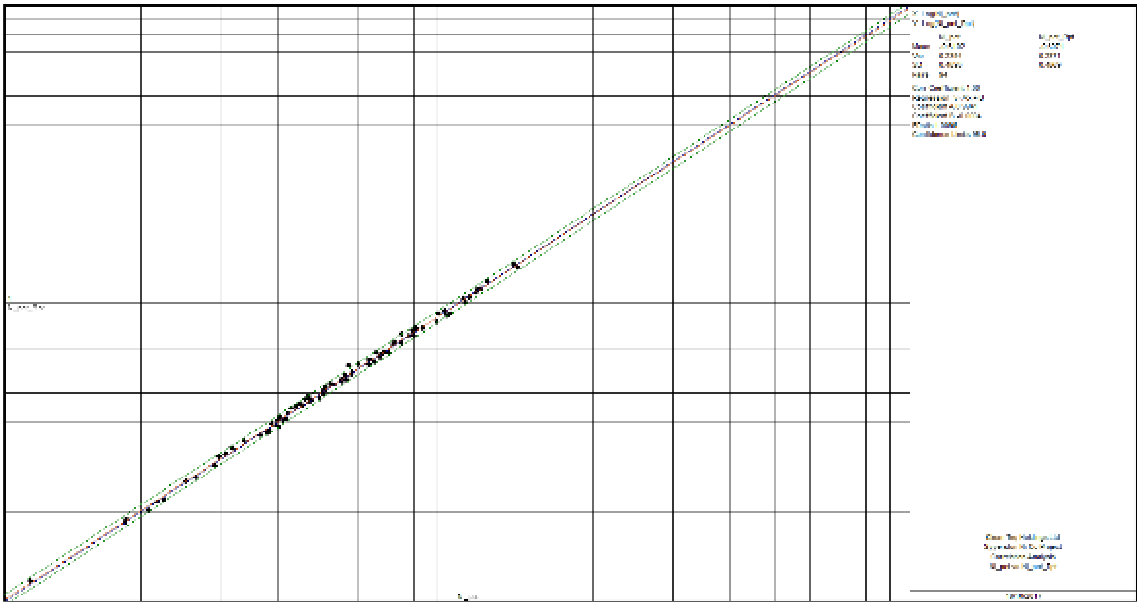


**Figure 11-39: SAC120 - SAC267 – Ni pct originals versusreplicates**

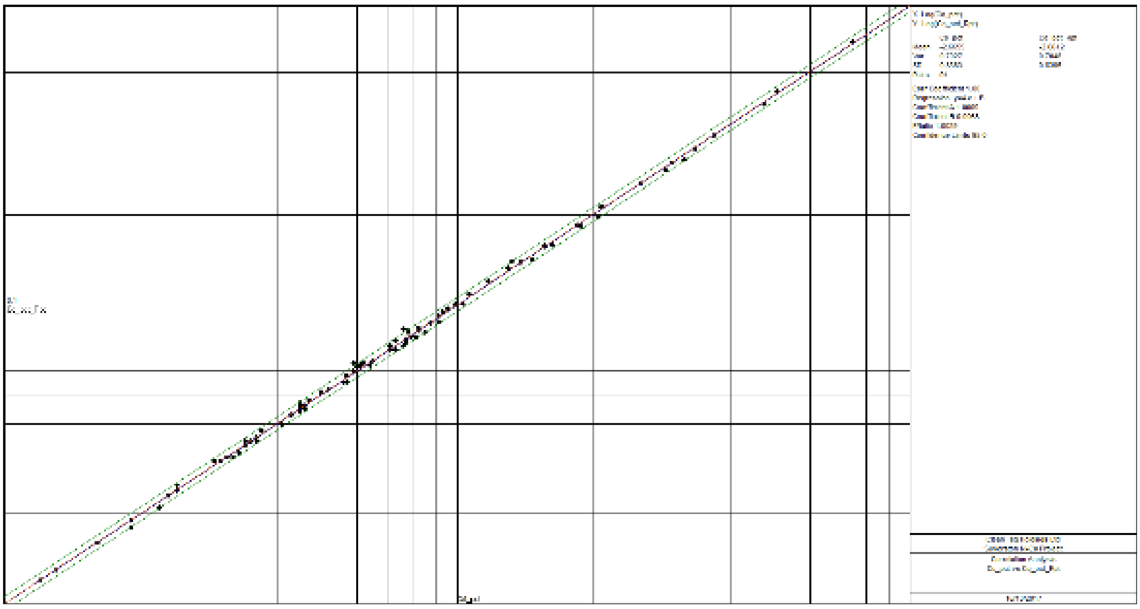


**Figure 11-40: SAC120 - SAC267 – Co pct originals versusreplicates**

## SRC001-SRC340 Drilling Program (August 1997-August 1998)

SRC001 – SRC340 Ni Replicate Samples:

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.8%
- Statistical status: Statistically ‘in-control’

SRC001 – SRC340 Co Replicate Samples:

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.8%
- Statistical status: Statistically ‘in-control’

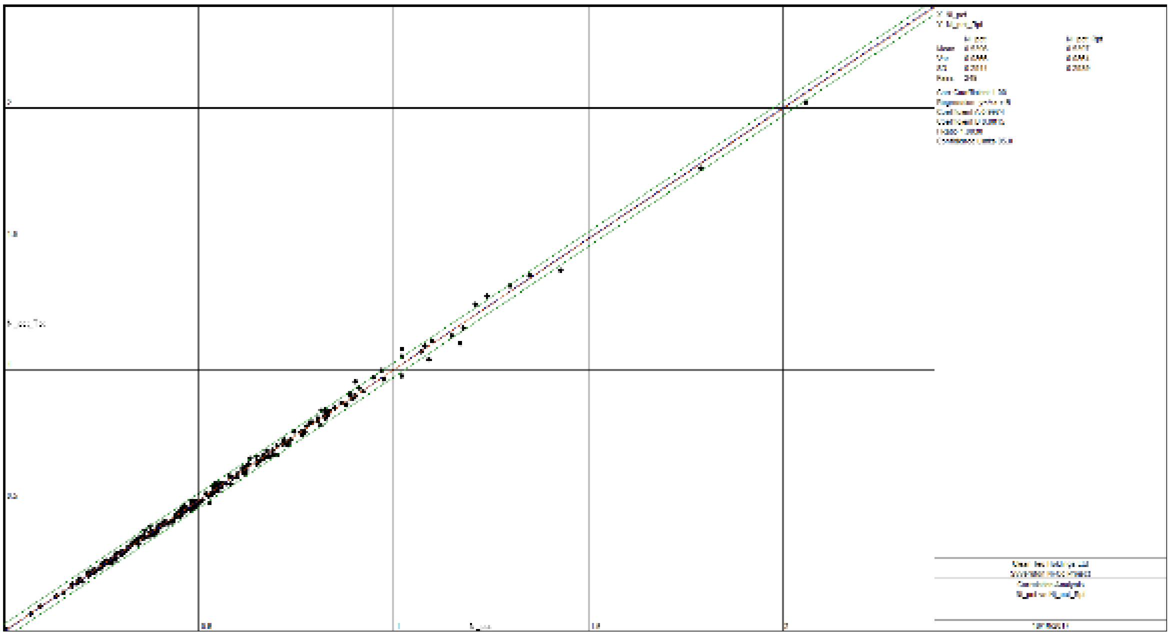


**Figure 11-41: SRC001 - SRC340 – Ni pct originals versusreplicates**

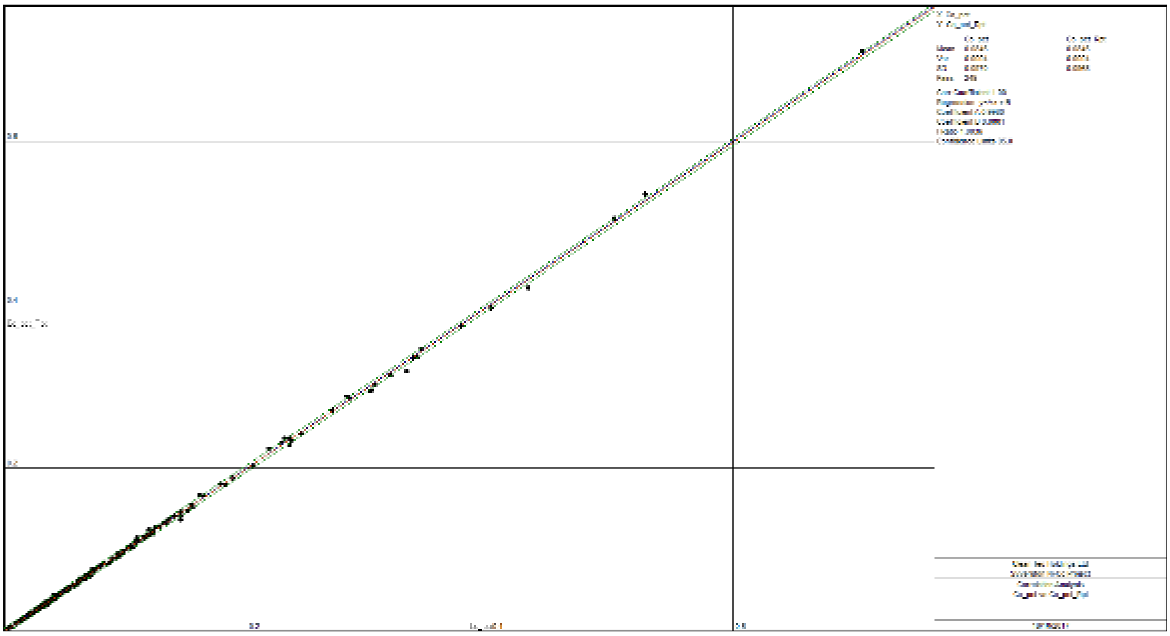


**Figure 11-42: SRC001 - SRC340 – Co pct originals versusreplicates**

## SRC0341-SRC1076 Drilling Program (August 1998-August 2000)

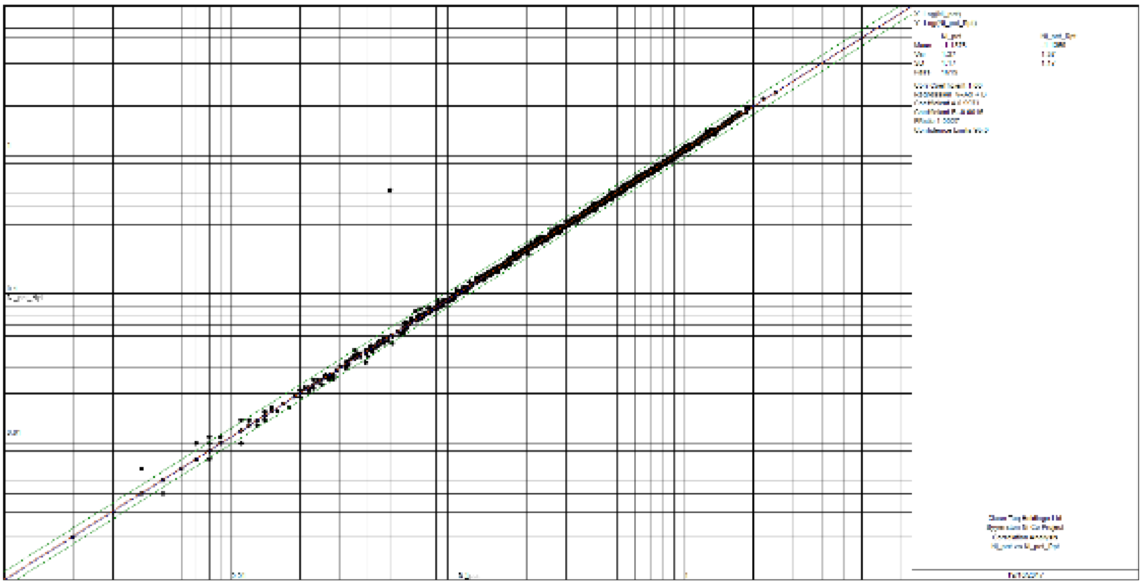


**Figure 11-43: SRC341 - SRC1076 – Ni pct original versusreplicates**

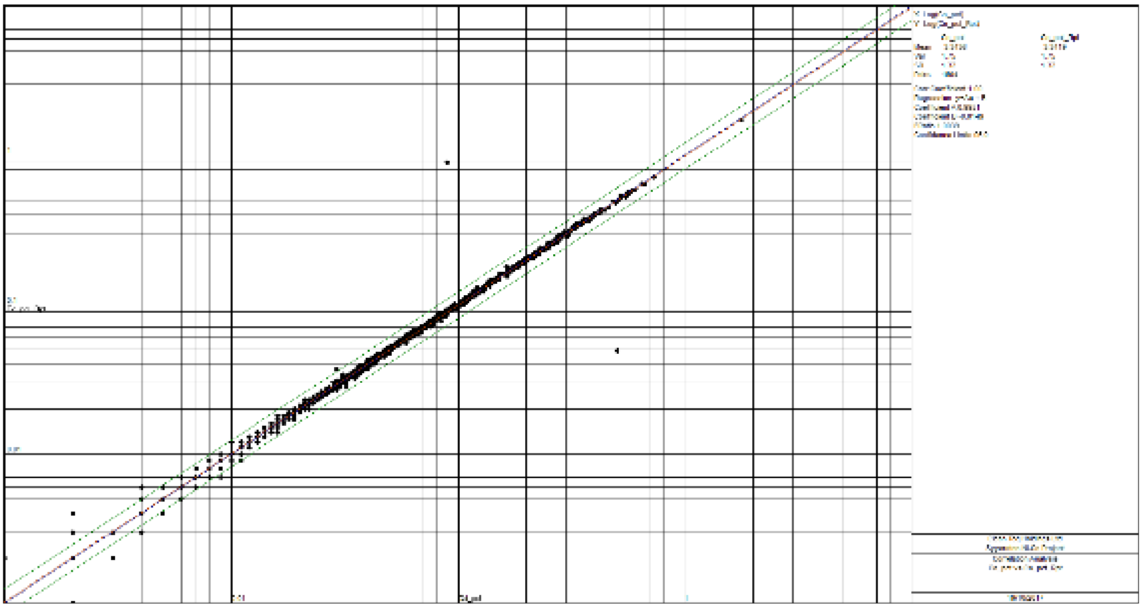


**Figure 11-44: SRC341 - SRC1076 – Co pct originals versusreplicates**

<u>SRC341 – SRC1076 Ni Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 4.8%
- Statistical status: Statistically ‘in-control’

<u>SRC341 – SRC1076 Co Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.0%
- Statistical status: Statistically ‘in-control’

## SRC1077-SRC1251 Drilling Program (February 2005-March 2005)

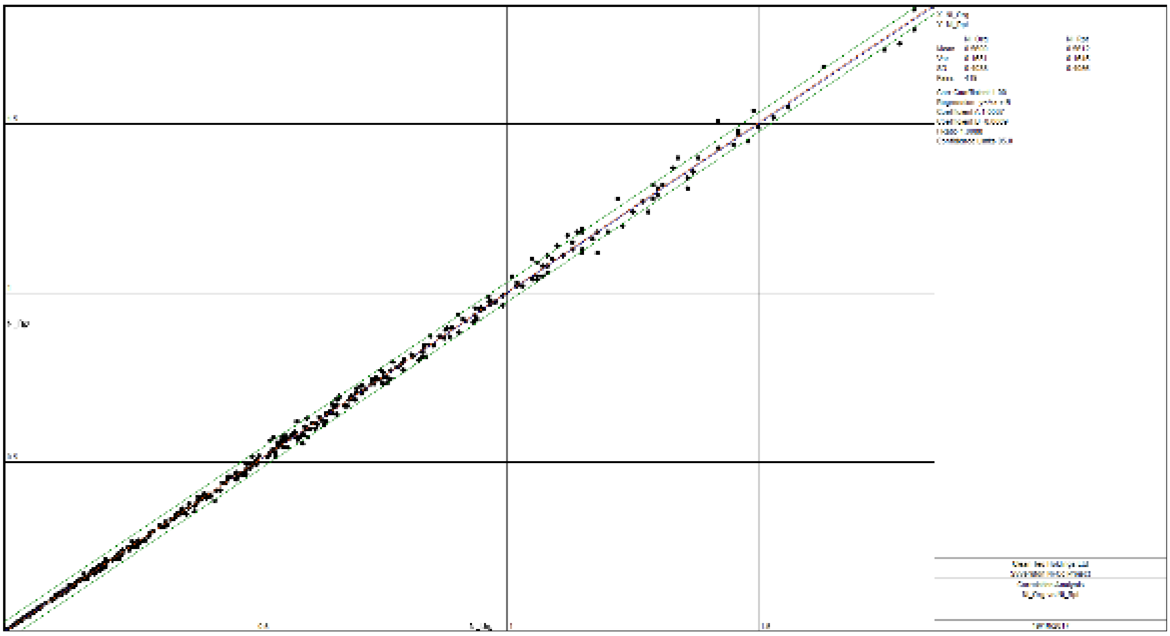


**Figure 12.34: SRC1077 - SRC1251 – Ni pct originals versusreplicates**

<u>SRC1077 – SRC1251 Ni Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: No bias
- ±10% QA/QC limits: 0.0%
- Statistical status: Statistically 'in-control'

<u>SRC1077 – SRC1251 Co Replicate Samples:</u>

- Correlation Coefficient: 1.00
- Diagonal versus regression lines: Very slight bias towards higher values
- ±10% QA/QC limits: 4.1%
- Statistical status: Statistically 'in-control'

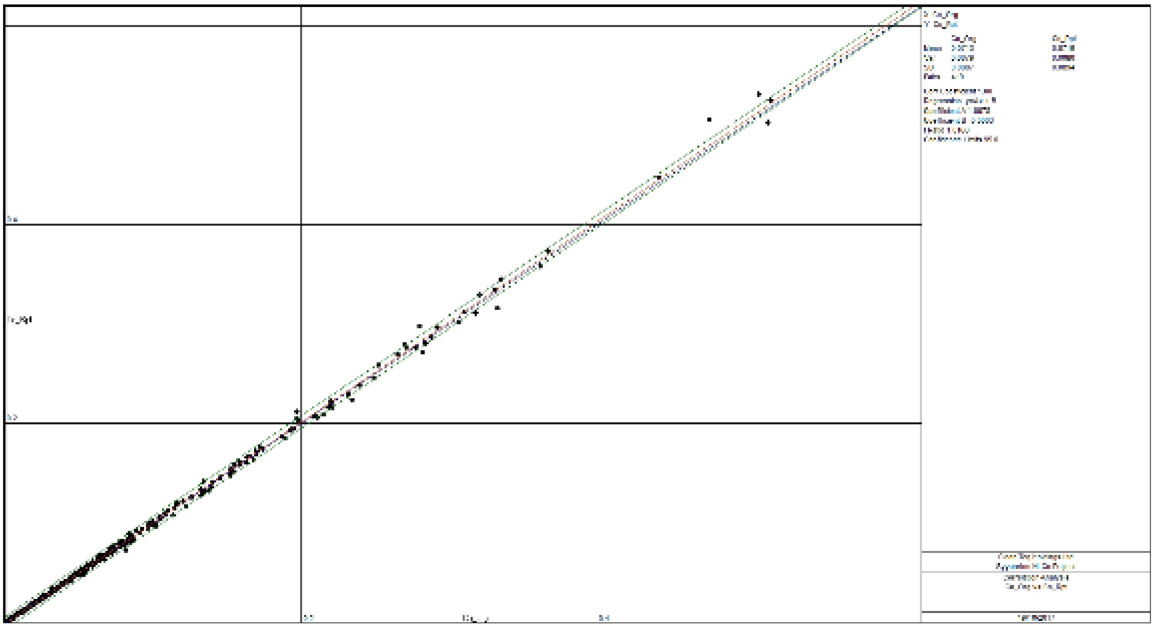


**Figure 11-45: SRC1077 - SRC1251 – Co pct originals versus replicates**

<u>SRC1263-SRC1276 Drilling Program (August 2014)</u>

ALS laboratory did not undertake any repeat assays whilst analysing the samples for this program and therefore, no replicate values are available to compare with the original values.

<u>SRC1277-SRC1310 Drilling Program (April 2015)</u>

ALS laboratory did not undertake any repeat assays whilst analysing the samples for this program and therefore, no replicate values are available to compare with the original values.

<u>SRC1311-SRC1368 Drilling Program (Nov2015)</u>

ALS laboratory did not undertake any repeat assays whilst analysing the samples for this program and therefore, no replicate values are available to compare with the original values.

### 11.8.5 Check assay programs ALS vs trace

### SAC120 – SAC267 Drilling Program (August 95-August 96)

<u>SAC120 – SAC267 Ni ALS versus Ultra Trace Check Samples:</u>

- Correlation Coefficient: 0.96
- Diagonal versus regression lines: Significant bias towards ALS routine values
- QA/QC limits: ±10% - 19.9%; ±20% - 1.4% and 30% - 0%
- Statistical status: Statistically 'out-of-control'.

<u>SAC120 – SAC267 Co ALS versus Ultra Trace Check Samples:</u>

- Correlation Coefficient: 0.96.
- Diagonal versus regression lines: Very slight bias towards higher check values
- QA/QC limits: ±10% - 40.0%%; ±20% - 11.2% and 30% - 2.9%
- Statistical status: Statistically 'out-of-control'.

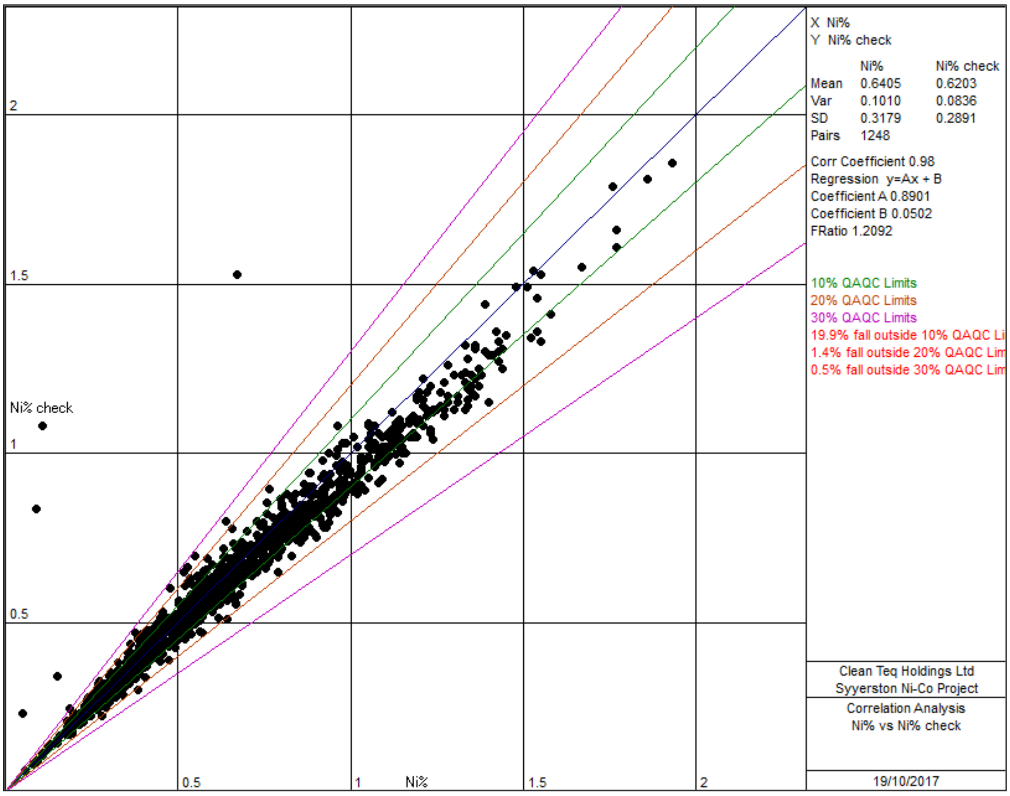



**Figure 11-46: SAC120 - SAC267 – Ni pct ALS versus Ultra Trace check assays**

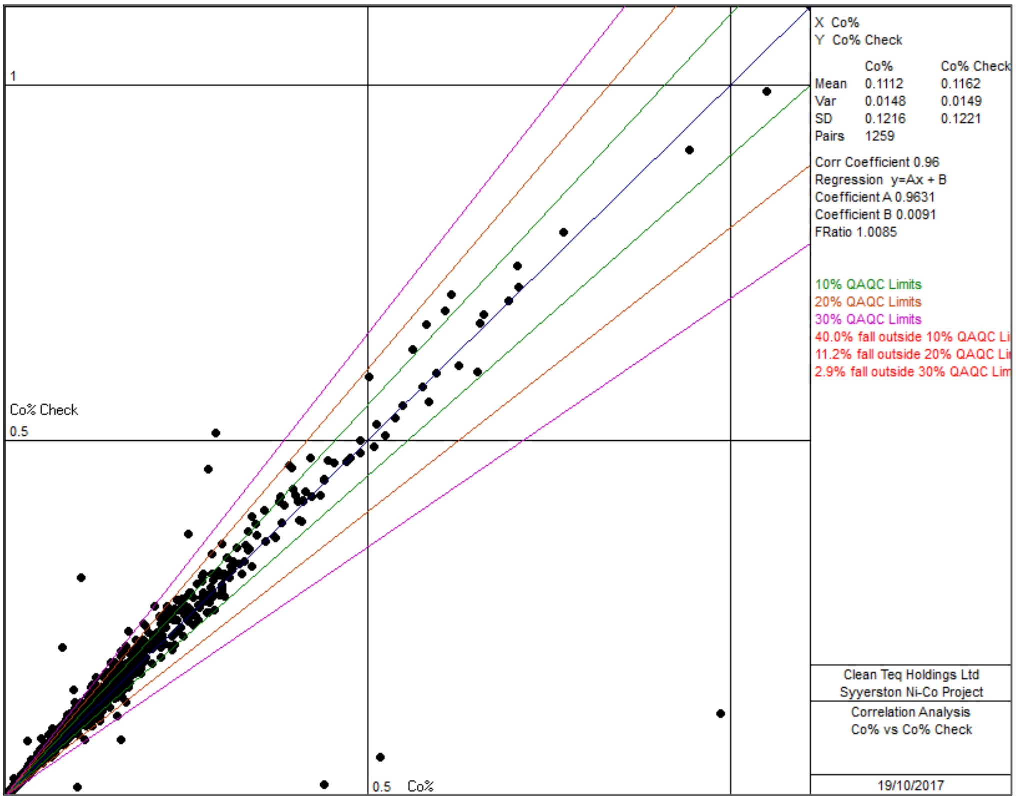



**Figure 11-47: SAC120 - SAC267 – Co pct ALS versus Ultra Trace check assays**

## SRC001-SRC340 Drilling Program (August 1997 – August 1998)

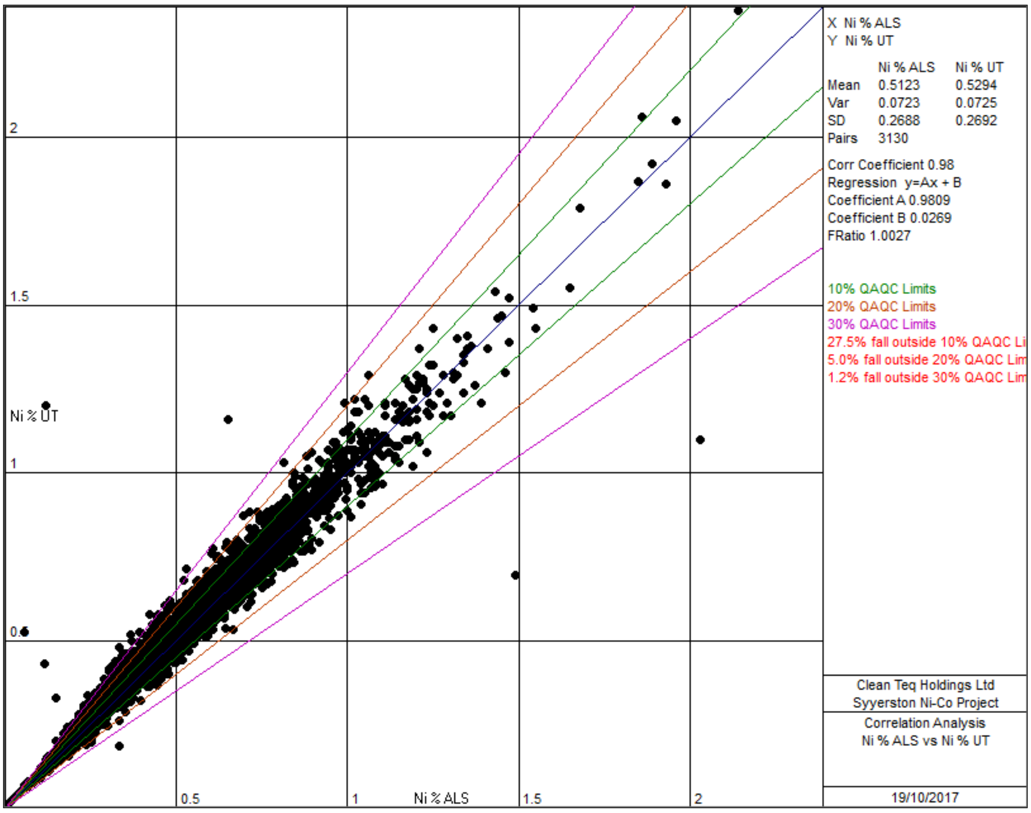



**Figure 11-48: SRC001 - SRC340 – Ni pct ALS versus Ultra Trace check assays**

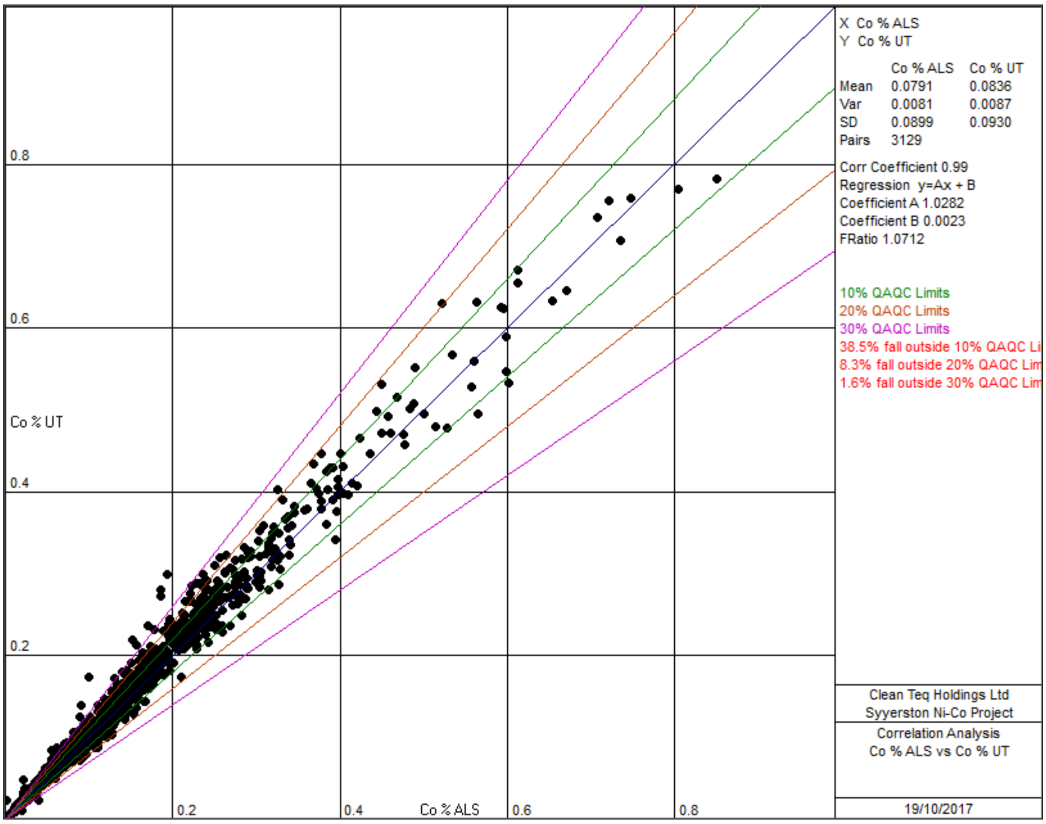



**Figure 11-49: SRC001 - SRC340 – Co pct ALS versus Ultra Trace check assays**

SRC001 – SRC340 Ni ALS versus Ultra Trace Check Samples:

- Correlation Coefficient: 0.96
- Diagonal versus regression lines: Significant bias towards ALS routine values
- QA/QC limits: ±10% - 27.5%%; ±20% - 5.0% and 30% - 1.2%
- Statistical status: Statistically 'in-control'.

SRC001 – SRC340 Co ALS versus Ultra Trace Check Samples:

- Correlation Coefficient: 0.96
- Diagonal versus regression lines: Very slight bias towards higher values
- QA/QC limits: ±10% - 38.5%%; ±20% - 8.3% and 30% - 1.6%
- Statistical status: Statistically 'out-of-control'.

## SRC0341-SRC1076 Drilling Program (Aug98-Aug00)

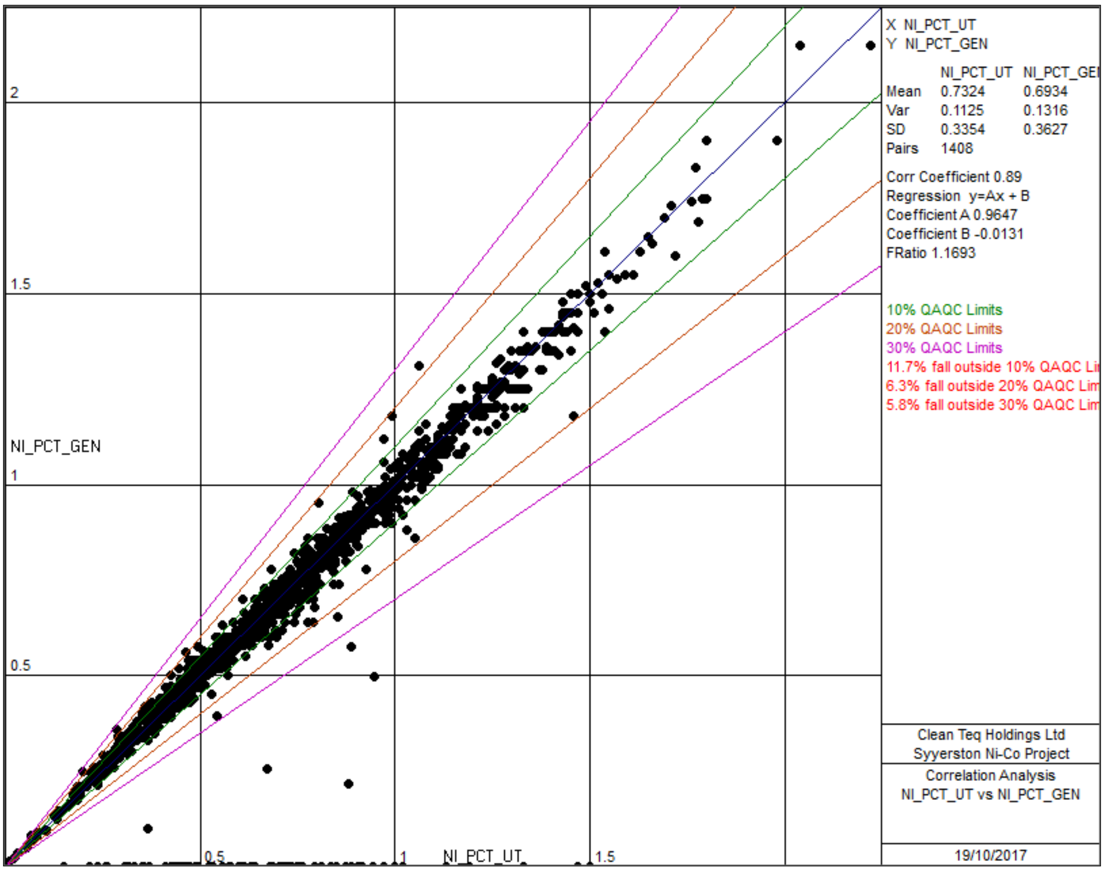



**Figure 11-50: SRC341 - SRC1076 – Ni pct ultra trace versus Genalysis check assays**

SRC341 – SRC1004 Ni Ultra Trace versus Genalysis Check Samples:

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: Significant bias towards ALS routine values
- QA/QC limits: ±10% - 11.7%%; ±20% - 6.3% and 30% - 5.8%
- Statistical status: Statistically 'out-of-control'.

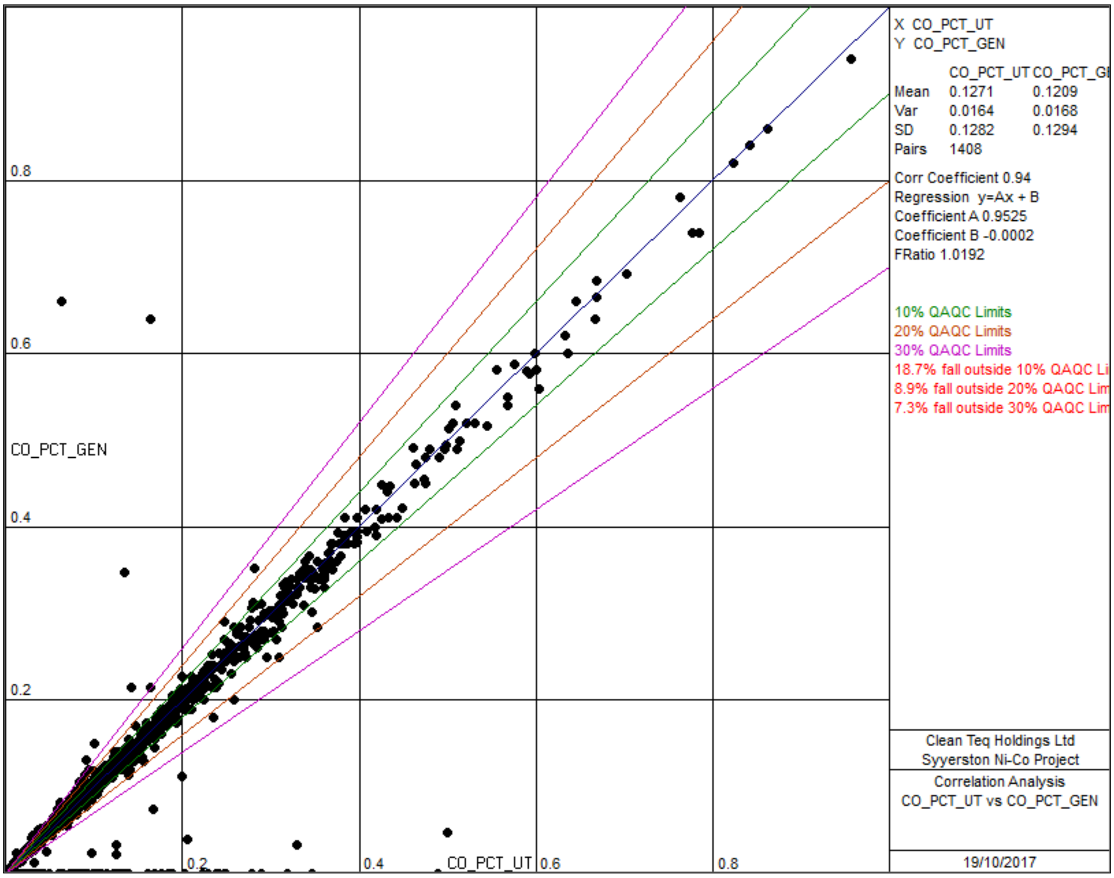



**Figure 11-51: SRC341 - SRC1076 – Co pct ultra trace versus Genalysis check assays**

<u>SRC341 – SRC1076 Co Ultra Trace versus Genalysis Check Samples:</u>

- Correlation Coefficient: 0.99
- Diagonal versus regression lines: Significant bias towards ALS routine values
- QA/QC limits: ±10% - 18.7%%; ±20% - 8.9% and 30% - 7.3%
- Statistical status: Statistically 'out-of-control'.

### 11.8.6 QAQA procedures comments

#### Shewhart Quality Control Charts

<u>SRC001-340; SRC341-1076 and SRC1077-1251 Drilling Programs</u>

These drilling programs used the Gannet SYS1 to SYS5 standards to monitor the assay quality control. They were submitted under various guises but this resulted in approximately 5 standard values per batch.

The assay quality control charts display moderate variability with the laboratory values within the range of 2SD's above the nominal value for the earliest drilling program and then in the latter programs the laboratory values are consistently between the nominal value and 1SD.

There is evidence of no trends in the earliest drilling program, however, the later drilling programs displayed a slight upwards trend towards the end of the drilling campaigns suggesting that quality control was not as stringent as in the earlier batches.

<u>SRC1263-1276; SRC1277-1310 and SRC1311-1368 Drilling Programs</u>

These drilling programs used the OREAS 45e standard only to monitor the assay quality control. The standard was submitted under a unique sample number with 1 standard per batch.

The Ni standard values for these programs varied in the range of +1SD to -1SD, whereas the Co standard values generally were in the range of -1SD to the nominal value. The SRC1277-1368 were drilled to delineate the Sc resource and another analyte was included in the QA procedures where the standard value in the SRC1277-1310 holes was observed in the -1SD to nominal value range, whilst in the later holes the standard Sc values generally ranged from-1SD to -2SDs.

Overall the QAQC results for the SRC001 to SRC1368 drill holes series are acceptable in terms of their reliability but on occasions there was an opportunity for improvement if the data had been rigorously monitored. As the assay data is predominantly historic then considering the emphasis placed on QAQC during this period, the records demonstrated that at least there was some level of control and the assay results were monitored.

## Duplicates

Duplicate sampling programs were carried out on the SRC341-1076; SRC1263-1276; SRC1277-1310 (1m and 2m samples) and the SRC1311-1368 drilling programs.

SRC341-1076 Drilling Program

The three parameters used to evaluate the duplicates *viz*. correlation coefficient, bias as measured by the regression line relative to the diagonal and the QAQC limits were all satisfactory for this program.

SRC1263-1276 Drilling Program

The correlation coefficients for Ni and Co were recorded as 0.98 and 0.99 respectively. There was a slight bias towards the duplicates. Whereas, the Ni values were within the ±10% QA/QC limits, the Co values reported 15.4% of the values reporting falling outside the QA/QC limits. The Co values were statistically ‘out-of-control’.

SRC1277-1310 Drilling Program

In the 2m sampling program the Ni, Co and Sc correlation coefficients were satisfactory at 0.99, 0.98 and 0.93 respectively. For the Ni and Co values there was a slight bias towards the duplicates at the lower values. The Sc values displayed a slight bias towards the original across the entire range. The Ni values were within the ±10% QA/QC limits but the Co and Sc values were excessive at 19.7% and 16.7% respectively and these results were not necessarily caused by ‘outliers’.

The 1m duplicate sampling program passed all three evaluation criteria with 0.99 correlation coefficients, no apparent bias and all three analytes were within the ±10% QA/QC limits.

SRC1311-1368 Drilling Program

The duplicate sampling for this hole series indicated that the correlation coefficients and bias criteria were met with no problems, the ±10% QA/QC limits were exceeded by 47.4%, 74.1% and 51.8% for Ni, Co and Sc respectively. This indicated that for all three analytes that they were statistically ‘out-of-control’. The duplicate sampling procedure required reviewing.

## Replicates

SAC120-267; SRC001-340; SRC341-1076 and SRC1077-1251 Drilling Programs

There were no observed problems with the replicate data using the three evaluation criteria viz. correlation coefficient, bias and ±10% QA/QC limits. All the programs indicated correlation coefficients of 1.00, whereas there was no bias with the exception of the SRC1077-1251 hole series where there was a slight bias towards the higher values for the repeats.

SRC1263 -1276; SRC1277-1310 and SRC1311-1368 Drilling Programs.

No records of ALS replicates were available for these programs to evaluate.

### Check Assays – ALS vs Ultra Trace

SAC120-267 Drilling Program

The original ALS values were checked by re-assay by Ultra Trace. The three criteria used to evaluate the duplicate and replicate data sets were correlation coefficient, bias and ±10% QA/QC limits.

The correlation coefficients for both Ni and Co were 0.96. The Ni values display a slight bias towards the routine values, whilst the Co values are slightly biased to the check assay values.

The ±10% QAQC limits indicates that 19.9% of the Ni values and 40.0% of the Co exceed these limits. Assessing the data set using the regression function and the 95% confidence limits, there are only 19 values or 1.4% 'outliers' for Ni, whilst for cobalt there are 17 values or 1.7% 'outliers'.

Statistically, if the 'outliers' are removed then the data set would be 'statistically 'in-control'.

SRC001-340 Drilling Program

The original ALS values were checked by re-assay by Ultra Trace. The three criteria used to evaluate the duplicate and replicate data sets were correlation coefficient, bias and ±10% QA/QC limits.

The correlation coefficient for both Ni and Co were 0.96. The Ni values display no bias between the ALS and Ultra Trace data, however, the Co values display a slight bias towards the Ultra Trace check assays.

Applying the ±10% QA/QC limits criteria, 27.5% of the Ni values and 38.5% of the Co exceed these limits. It is considered that if the 'outliers' were removed, then the data set would be statistically 'in-control'.

### 11.8.6.1.1 SRC341-1076 drilling program – UT versus Genalysis

The original Ultra Trace values were checked by re-assay by Genalysis. The three criteria used to evaluate the duplicate and replicate data sets were correlation coefficient, bias and ±10% QA/QC limits.

The correlation coefficient for Ni was 0.96 and 0.99 for Co. The data set indicates that there is a slight bias in the higher values towards the Ultra Trace results for both Ni and Co.

The application of the ±10% QA/QC limits criteria indicates that 11.7% of the Ni values and 17.7% of the Co values exceed these criteria. If the 'outliers' were removed, then the data set would be statistically 'in-control'.

## 11.9 Independent custody sampling – 2017 Mineral Resource estimate

A separate program of independent custody sampling has been conducted for the 2017 Mineral Resource Estimate.

A twin RC and diamond drill hole program was designed to test 10 RC drill holes from the historic resource definition drilling. A twin RC hole was positioned approximately 5m diagonally from the previous RC hole in the NE quadrant, whilst a diamond hole was to be drilled in a similar location in the SW quadrant.

The holes were positioned to provide a spatial spread between the eastern and western mineralised zones and to optimise the intervals of the GZ and SGZ within the planned holes.

The holes were drilled adopting the current sample protocol and logging procedures and were observed by an experienced sample technician to record the procedures and report on behalf on the Qualified Person.

This process was fully documented to identify a 'chain-of-custody' for the sampling procedures and was followed through to the delivery of the samples at the routine laboratories sample preparation facility, where the technician ensured that during the procedures, the samples were under scrutiny of responsible personnel at all stages.

The sample interval data was compiled by the Senior Geologist on site and on receipt of the assay results files, was forwarded to the Qualified Person for comparison with the historic data.

The hole location, survey and sample data were compiled as load files for the drill hole database directly from the site geologist's records. They were imported into the drill hole database together with the laboratory assay file. Subsequently, the collar, survey and assay files were exported from the database in Micromine format.

The twin hole data was compiled in Micromine and presented as graphic logs with the following fields:

- Depth
- Ni% graph
- Ni% value
- Co% graph
- Co% value
- Fe% graph
- Si% graph
- Al% graph
- Latzone hatch
- Ni and Co% interval.

The mineralised intervals were calculated in the graphic log function by applying a 0.4% Ni 'cut-over' value. These mineralised intervals are reproduced in Table 11-16.

**Table 11-16: Custody sampling – original vs twin hole intervals (0.4% Ni cut-over value)**

| Original Hole | Original Hole Intersection | Twin Hole | Twin Hole Intersection |
|---|---|---|---|
| SRC946 | 25m @ 0.91% Ni & 0.111% Co | SRC1418 | 30m @ 0.86% Ni & 0.088% Co |
| SRC890 | 22m @ 0.79% Ni & 0.119% Co | SRC1419 | 21m @ 0.76% Ni & 0.095% Co |
| SRC742 | 31m @ 0.83%Ni & 0.113% Co | SRC1420 | 25m @ 0.87% Ni & 0.168% Co |
| SRC767 | 28m @ 0.64% Ni & 0.076% Co | SRC1421 | 26m @ 0.67% Ni & 0.076% Co |
| SRC774 | 19m @ 0.65% Ni & 0.110% Co | SRC1422 | 17m @ 0.52% Ni & 0.172% Co |
| SRC647 | 21m @ 0.67% Ni & 0.127% Co | SRC1423 | 18m @ 0.47% Ni & 0.068% Co |
| SRC1125 | 35m @ 0.75% Ni & 0.085% Co | SRC1424 | 31m @ 0.83% Ni & 0.051% Co |
| SRC445 | 31m @ 0.78% Ni & 0.156% Co | SRC1425 | 30M @ 0.78% Ni & 0.149% Co |
| SRC556 | 14m @ 0.59% Ni & 0.117% Co | SRC1426 | 25m @ 0.48% Ni & 0.116% Co |
| SRC597 | 17m @ 0.55% Ni & 0.092% Co | SRC1427 | 19m @ 0.62% Ni & 0.144% Co |

An assessment of the twin hole data was undertaken and the key variables are summarised as follows:

- Average interval difference -0.10 m
- Average Ni% difference -0.03%
- Average Co% difference 0.0%
- Average % difference interval 4.0%
- Average % difference Ni% -4.5%
- Average % difference Co% 2.9%.

These results indicate that there are minor differences in the interval meters, Ni values and Co values, where all three variables reported less than a 5% difference. It is considered that this exercise has provided an ample validation check of the historic holes especially with respect to the sampling process and assay confidence. It also provides an adequate test for the 'chain-of-custody' procedural requirement.

# 12 Data Verification

## 12.1 Validation checks by SRK

SRK had access to the input data, working files and final resource model completed by McDonald Speijers in 2015. SRK reviewed the detailed technical reports by McDonald Speijers for the 2005 and 2015 resource estimate in detail and used them to form the basis of this current report in a merged, summarised and updated form.

SRK has knowledge of McDonald Speijers and its lengthy history of involvement in the mining industry in Australia and overseas; SRK has no reason to consider that the work and reports by McDonald Speijers are anything other than professional and to have been completed in accordance with both the Australian JORC Code and the Canadian CIM definition standards.

SRK used the drilling database as supplied by McDonald Speijers and independently reproduced the resource tonnages and grades at 0.6% NiEq cut-off to within 10%.

In SRK's opinion, the robustness of the resource tonnages and grades is demonstrated by the negligible changes in the resource tonnage and grades over the last three resource estimates (1999, 2005 and 2015), with additional drilling included in each update. The multiple phases of drilling and the different owners and the different laboratories used all point to confidence in the robustness of the overall resource.

SRK's Qualified Person for Mineral Resources, Danny Kentwell, has experience with other Ni-Co laterite deposits similar to the described and modelled Ni-Co mineralisation of the Syerston deposit.

Peter Fairfield (SRK) visited site in June 2017; however, it was not possible to observer active drilling or exposed mineralisation as there were no current exploration or mining activities taking place. Drill collars and drill pads have largely been re-habilitated and little evidence of them remains.

Verification checks completed by McDonald Speijers for its 2005 NI 43-101 Technical Report for Ivanplats Pty Ltd are described in the following section. SRK (Danny Kentwell) has completed spot checks on the database and examined working files used by McDonald Speijers for validation, and is satisfied that the validation checks and conclusions are correct as described. As the Qualified Person for the Mineral Resources, Danny Kentwell takes responsibility for Chapters 6 to 12, and 14, of this Report.

## 12.2 Validation checks by McDonald Speijers – 2005

### 12.2.1 Drill hole database files

McDonald Speijers understands that when Ivanplats acquired the Project, the Administrators of Black Range were unable to supply comprehensive digital data records. A drill hole database was acquired (courtesy of Andrew Spinks) and supplied to SRK on 19 November 2004. This was reported to contain data for all drill holes up to December 2002, when fieldwork on the Project apparently ceased.

On 10 April 2005, Ivanplats supplied drill hole data for additional infill holes up to SRC1251, as a set of Excel files. Assay results were only available up to and including SRC1193. Due to time constraints, the database was frozen at that date for the purpose of resource estimation. Outstanding holes consisted of some additional RC twin holes around old Calweld holes and some closer-spaced holes around previous Pt intercepts.

### 12.2.2 Drill hole data checks

McDonald Speijers conducted a number of checks on the internal consistency and integrity of the drill hole data files. These included checks for:

- Duplicated collar coordinates
- Consistency of numbers of holes and hole numbers between different file types
- Overlaps, reversals or gaps in From-To intervals in assay and geology files
- Numerical values obviously outside realistic limits
- Listings of alphanumeric codes and comparison with available legends.

#### Collars

The final, updated collar file contained 1,627 records for 1,627 holes.

There were 14 pairs of duplicated collar coordinates in the old data. While the coordinates should not have been absolutely identical, they all appeared to be valid cases of aircore or RAB holes (c.1994) replacing earlier RAB holes (c.1988-89), so no data corrections were attempted. The RAB holes were not used in resource estimation.

The new data contained one pair of near-coincident collar coordinates. This appeared to be a survey error resulting from an old diamond drill hole collar being picked up instead of one of the recent RC twins of Calweld holes (SRC1247). No assays were available for the new hole and it was not used in resource estimation. Ivanplats need to correct its collar coordinates in its database.

Four collars that lacked surveyed elevations were assigned values based on the adjacent holes. These should be reasonable approximations.

#### Survey

The down hole survey file (dh_sur.dm) contained 3,254 records for 1,627 holes. Hole numbers matched those in the collar file.

Most holes were vertical and unsurveyed, so the vast majority of entries were notional end of hole values created during data transfer.

#### Assays

The final assay file contained a total of 39,300 records for 1,562 holes. There were 65 holes present in the collar file that were not present in the assay file. These consisted of the seven old holes and the last 58 RC holes drilled in 2005 (SRC1194-SRC1251) for which assays had not been received.

McDonald Speijers suspect that holes P30 and P36 may be phantom entries left over from when the old 'P' series of holes were renamed as the 'FR' series. The others were triple-tube diamond drill holes, which were apparently not sampled and assayed.

There were no overlaps or reversals in From-To intervals in the assay files.

#### Assay types and units

Most field names in the old assay file incorporated an indication of the originating laboratory. The old assay data were consequently imported in two stages based on the laboratory involved - ALS data (raw_als.dm) and Ultratrace data (raw_ut.dm). The 2005 data all originated from Ultratrace and were initially loaded to the file 'new_ut.dm'.

In the old assay file, most elements were reported in two separate fields containing values in ppm and in %, so the Datamine assay files were split into two sets to facilitate checks and comparisons. All % values from both laboratories were compiled into assays%.dm and values in ppm were compiled into assaysp.dm.

All nickel and cobalt values shown in ppm were matched by an equivalent % value, but some values in % did not have a matching ppm record. McDonald Speijers presume that assays were generally reported and entered in ppm, with % values subsequently calculated from the ppm values, with a minority of cases where values were originally reported and entered only in %.

Only Au, Pd and Pt values were stored without reference to the originating laboratory. However, field names distinguished fire assays from Aqua Regia digests. No Aqua Regia values occurred together with Ni ppm results shown as being from ALS, indicating that fire assays were probably all from ALS and Aqua Regia/AAS results were probably all from Ultratrace.

The old and new assay data from both laboratories was finally combined into a single file (dh_ass.dm) containing accepted values for each metal and a flag indicating the laboratory that provided the nickel and cobalt values. Where there was any duplication, Ultratrace assays were accepted in favour of the older ALS results.

#### Geology

The geology file (dh_geol.dm) contained 37,938 records for 1,619 holes. Only eight holes present in the collar file were not present in the geology file. These were all old RAB holes, apparently dating from 1989, that were not used in resource grade estimation.

There were no overlaps or reversals in From-To intervals.

As is often the case with geological coding systems, there were very large numbers of unique codes in most fields, far more than indicated in the geological legends.

#### Density

The old database file contained 18,498 records for only 107 holes. Five of the holes were not present in the assay or geology files (SDD6, 8-10,12).

The file only contained partial down hole gamma probe data for diamond drill or RC holes. Other data located elsewhere was subsequently added. No density values from Calweld holes or from physical measurements on drill core appeared to be present. These were also added later from csv files located in old working directories.

The initial file contained a mixture of wet and dry density values, apparently all derived from down hole gamma logging. Some results were for 10 cm intervals and some for 1 m intervals down the holes. For working purposes McDonald Speijers composited 10 cm data over 1 m intervals to match normal assay intervals.

The final, collated and composited density data file (dens.dm) contained 5,199 records for 148 holes.

#### Other data

No sample recovery or weight data could be located in the old database. Weights for sample splits subsequently received at the laboratory were located in other files and added later.

Core recovery values for diamond drill holes were also missing. This information was recovered from original logs for holes SDD6 to SDD13, but logs for SDD1 to SDD5 did not include core recoveries. Available core recoveries were averaged over assay sample intervals before being merged into working data files.

### 12.2.3 Drill hole data validation

#### Previous validation checks

In the course of the previous feasibility study, SLA conducted validation checks on a selection of drill holes scattered across the deposit.

Assay values were reportedly checked for:

- 39 pre-June 1999 aircore or RC holes
- 27 post-June 1999 RC holes.

Assays in the database were listed and compared with original laboratory reports. In pre-June 1999 cases where samples had been assayed at both ALS and Ultratrace, only the Ultratrace values were checked because these were the values used in resource estimation.

SLA reported that no discrepancies were found apart from a few erroneous Si values in the first one or two samples of RC holes drilled in January 1999. It was stated that these were subsequently corrected.

SLA also reported that they checked all samples with Ni values greater than 1.8% or Co values greater than 0.75% (178 samples) against original laboratory reports and detected no errors.

Reports also indicated that SLA checked collar coordinates for 153 pre-June 1999 RC holes and 27 post-June 1999 RC holes. This apparently involved:

Transformation of database local grid coordinates to AMG using a transform given in a March 1999 report from Terra Sciences.

Checking against "original" survey listings of AMG coordinates.

No problems were reported for the post-June 1999 holes. But for the pre-June 1999 holes the following discrepancies were noted:

There were four sets of duplicated coordinates in the original survey listing. It appears that these were taken to be errors in hole identification in the field by the surveyors. This implies that there may have been at least 4 holes that were not picked up by the surveyors.

There were 20 sets of coordinates in the original survey listings without valid hole numbers. These were apparently assigned hole numbers by Black Range based on their records of drill hole locations. This was probably reasonable.

No original survey records were located for aircore holes.

Overall, it seemed that the SLA checks had indicated that the collar survey data were probably reliable, but it seemed that supporting survey records were incomplete or imperfect.

### 12.2.4 Additional validation checks

As part of our initial review phase McDonald Speijers selected a total of 76 drill holes from the database for validation spot checks.

RAB holes were excluded from the exercise because they were not acceptable for use in resource estimation. Diamond drill holes and Calweld holes were also excluded because of their small numbers, but their collar coordinates were verified.

The holes were selected to be both widely spread over the deposit and from different time periods. No attempt was made to either include or exclude holes previously checked by SLA.

A few additional holes were included because visual examination of sections or comparisons of twinned holes had indicated that they compared poorly with adjacent holes, either in terms of logged geology or assay results.

The checks conducted were:

- Collar coordinates were compared with original field logs and survey reports
- Assays were compared with original (hard copy) laboratory reports.

After receipt of data files for the 2005 infill drilling, an additional set of spot checks were conducted on the new holes. All collar coordinates were checked against proposed hole locations and survey reports. Assay data for 24 randomly selected holes were checked against original electronic copies of assay reports received by us directly from the laboratory. These checks showed no significant discrepancies apart from the collar coordinates for 1 hole.

## Collar coordinates

RC holes collar coordinates agreed with those in survey reports where these could be found. No survey reports could be located for aircore holes.

In the absence of survey reports, coordinates were checked against those recorded on the original logs. Discrepancies of up to 50 metres were noted and in 17 cases differences exceeded 10 m. It may have been that a significant proportion of holes were drilled well away from the mark-out pegs for various practical reasons, but McDonald Speijers would not have expected these offsets to have been more than a few metres.

The collar coordinates of the drill holes, particularly the aircore holes, could not be satisfactorily validated. The quality of the survey database is open to some doubt and there could be significant errors in the collar locations of some holes.

The coordinates for one recent RC hole (SRC1247) did not agree with its planned location. It appeared that an old DDH collar had been picked up in error. This hole was not used in resource estimation.

## Assays

McDonald Speijers checked 1,673 old assay records against original, hard copy laboratory assay reports for Ni, Co and Pt, and for Au, Mg, Fe, Zn, Cu, Al, Cr, Ca, Sc, Si and Mn where these were present. McDonald Speijers located assay reports for all but 5 of the samples.

No significant errors were located.

McDonald Speijers also checked 908 assay records from the 2005 drilling for all elements reported. No discrepancies were found. The new assay data file appeared to be of good quality.

## Geological Codes

In the course of a short initial site visit, McDonald Speijers examined bagged reject from a number of holes in relation to print outs of assays and geological codes from the drill hole database.

Many of the bags were in too poor a condition to be systematically examined and the procedure was curtailed by rain. In the time available, McDonald Speijers was able to scan only six holes (SRC426, 500, 560, 880, 901, 923). In all cases the physical characteristics of the rejects, and the depths at which distinct changes were observed, were consistent with changes in the assay data, particularly for elements such as Fe, Si and Mg. However, in three out of the six cases (SRC500, 880, 923) there were some significant and obvious contradictions with the geological codes. Subsequent checks showed that the codes in the database were consistent with the original logs, so McDonald Speijers was forced to conclude that some of the field logging was of variable quality. The rate of error appeared to be disturbingly high, suggesting that unfortunately, geological codes have to be treated with caution.

In cases where they are contradictory it would be reasonable to allow interpretations to be guided by the assay data in preference to the geological codes.

McDonald Speijers observed some logging during the early stages of the 2005 program and gained the impression that it was of satisfactory quality. The greatest areas of uncertainty appeared to be in discriminating between transported alluvium and residual overburden and in identifying the boundary between the SGZ and the saprolite.

## 12.3 2015 data validation

McDonald Speijers was supplied with updated drill hole data files by OreWin who were preparing a separate Sc resource estimate. These files contained data for an additional 106 RC holes drilled in 2014-2015.

Because McDonald Speijers was aware that OreWin had conducted validation checks, and because the new data only amounted to about 6% of total metres in holes that McDonald Speijers accepted for use in resource grade estimation, apart from file integrity checks as outlined above McDonald Speijers took the new data at face value.

SRK has not completed specific checks on the 2015 data.

## 12.4 2017 data validation

### 12.4.1 2017 Geobank CLQGB drill hole database

In 2017, it was determined that a new drill hole database should be created and the Micromine Geobank program was selected. Geobank provides a 'front-end' to a data model, which resides in Microsoft SQL Server.

The Geobank model contains a series of tables, which enable the efficient storage of collar, survey, assay and geology data, with adequate provision to cater for a wide range of data fields to be stored in the various tables associated with the four principal data types i.e. collar, survey, assay and geology.

Some of the tables are 'stand-alone' tables e.g. GB_PROJECT, whereas others are 'related' and provide a 'compound' approach to the storage of that data type e.g. GB_SITE and GB_SITE_SURVEY, which are both associated with the collar data.

The data model has provision for using multiple coordinate systems i.e. Local, AMG84 and MGA by using tables defining the coordinate system and storing them as separate instances. The tables used for this purpose are as follows:

- GB_SYS_CS_DATUM
- GB_COORD_SYSTEM.

The survey data tables included the following:

- GB_DOWNHOLE_SURVEY_HEADER
- GB_DOWNHOLE_SURVEY_COORDS
- GB_DOWNHOLE_SURVEY_DATA.

The assay data including routine, composite and duplicate data are stored in a sample table aptly named GB_SAMPLE. The blanks and standards samples are recorded in a QA/QC table named GB_SAMPLE_QAQC.

The geology data is stored in a table entitled GB_LITHOLOGY, allowing lithology, mineralogy, colour, etc. to be recorded within the table.

Templates were created in Micromine to use in the populating of load files for the collar, survey, assay and geology data and exported as Excel files and finally saved as csv format files. The data records used in the load files were sourced either from the Black Range Minerals/Ivanplats 'Syerston_DB.mdb' Access database or other historic Ivanplats/Scandium21 data files. These data files were principally in Excel/csv format.

The load files contained field names and data types comparable with those in the tables within the Geobank model. Several fields required that entries for these values should be placed in 'parent' tables prior to populating the 'child' table and this was determined by a data loading order (DLO).

The csv load files were imported into Geobank using the 'ad hoc' import object, which enabled the fields within the load file to be matched with those in the data model table. The use of the same field names in the load files as those within the data model simplified the matching process between the load files and the model table fields.

To comply with the 'chain of custody' requirements, it was necessary to identify the original laboratory assay result files. These files were identified within the archive records and the integrity of the original files was maintained to ensure that the file date stamp remained as issued. To overcome investigating the file contents and editing the metadata if required, the files were copied and marked as a copy using a suffix to the filename.

The data model GB_ST_RESULT table required several mandatory metadata items including Despatch No., Lab Job No., Sample ID, Result, Elements, Units and Method. It recorded several additional metadata fields, which in the current import included Lab Date and Received Date.

A search for the original laboratory assay result files identified 67% of the original 'sif' or 'csv' files with most of the metadata present in the file header with generally only the Date Received value being added to the file copy. The balance of the assay results files required the metadata to be added as a header as they only contained the element and units within the metadata.

The Australian Laboratory Services (ALS) assay data for the SAC120 to SAC267 holes was exported from the Syerston_DB Access database and a compliant load file was created in Excel then converted to a csv load file.

Prior to importing the assay results into the ST_RESULT table, the composite and duplicate records were imported into the GB_SAMPLE table. The records for the standards and blank were similarly imported into the GB_SAMPLE_QAQC table.

Several assay results formats were created in Sample Tracker, the import object used to import laboratory assay results files into Geobank. Separate formats were required for both ALS and Ultra Trace results as well as for 'sif' and 'csv' variants for the file types.

A total of 166 assay report files from ALS and Ultra Trace were loaded into the Geobank database, which contained 39,546 routine assay records. In addition, 1,937 duplicate and 3,420 standard and blank assay records were loaded into the database.

### 12.4.2 Data validation 2017 Resource model

The collar and survey data files were compiled from the historic database (Syerston_DB) and other Excel/csv records and validated against the available Licensed Surveyor's reports.

A drill hole database compiled in Micromine from historic sample interval records merged with the original laboratory assay reports and used for the 2017 Resource Estimation process.

A 'high level' (Geobank) drill hole database was developed in parallel with the resource estimation process. Checks have been carried out comparing the two data sets and they are comparable.

The geology data file was developed totally from historic records and combined with the current (2017) geological interpretation for the LATZONE.

# 13 Mineral Processing and Metallurgical Testing

Extensive metallurgical piloting on the extraction and recovery of nickel and cobalt was completed by the two previous owners, including variability testing over 100 composites of different ore lithologies. This work has provided a solid basis to establish the design criteria for the Nickel Cobalt Project as well as the scandium processing option which is not part of the base case. During each of these testwork programs nickel and cobalt were the primary targets, with scandium also followed in the analysis. This work provides a relatively high degree of confidence on metal extraction using HPAL and subsequent unit process design criteria. In addition to this previous work, basic sighter tests were carried out to confirm the previous metallurgical test results.

The more recent metallurgical testwork and processing development focus of Clean TeQ was applying the RIP technology, a change to the original processing flowsheet, to the HPAL discharge slurries. Clean TeQ has not yet undertaken extensive testing of the flowsheet downstream of RIP, specifically the nickel and cobalt solvent extraction and the final nickel and cobalt crystallisation process, however, they are not considered to be a novel technology. Further discussion on the key processing technologies and the supporting metallurgical testwork are provided below.

## 13.1 Processing technology

### 13.1.1 High pressure acid leach

The nickel, cobalt and scandium bound within the goethite or clays can be released from the solid matrix by acid leaching. The soluble iron species subsequently hydrolyse to basic iron sulphates which, under the conditions present in the high-pressure acid leach (temperatures greater than 220°C), react to hematite and other iron products that are not soluble under these conditions. This regenerates acid, a major operating cost. Soluble aluminium sulphates also hydrolyse to the basic sulphate salts. Atmospheric leaching of laterite ores can also extract significant nickel and cobalt values, however, the significant leaching of nearly all the iron from the solids results in a high acid consumption and requires a high initial acid addition (the iron sulphates do not hydrolyse at this low temperature).

There is a high degree of confidence in the robustness of the HPAL process to extract nickel and cobalt from lateritic ores. Commercial high-pressure acid leaching of laterites commenced at Moa Bay, Cuba in the late 1950's. At Moa Bay, approximately 2 Mtpa of limonitic ore is processed by a treatment route consisting of pressure acid leaching in vertical brick lined autoclaves, solid-liquid separation in a multi-stage thickener circuit, neutralisation using coral mud and mixed sulphide precipitation with hydrogen sulphide gas.

The pressure acid leach route was further developed by Amax in the 1970's. They carried out an extensive pilot-scale test program to improve and further develop the process used at Moa Bay. Amax were the first to adopt multi-compartment horizontal autoclaves with mechanical agitation of each compartment. Although Amax never proceeded to commercial production, this work made a major contribution to the longer-term development of the technology.

Work on the Western Australian laterites was initiated in 1988 for the Bulong Project. Other Western Australian projects, including Anaconda Nickel's Murrin Murrin Project and Centaur Mining's Cawse Project, commenced development work in 1994 and 1995 respectively.

Bulong, Cawse and Murrin Murrin all made decisions to proceed with construction during 1995/1996. Commissioning of all plants commenced in the second half of 1998. Processing varies between the Western Australian producers influenced by the nature of the ores (how much sulphuric acid is consumed by alkaline minerals present) and the quality of available process water as well as the downstream processing route (post HPAL) selected and final products as there are several options.

Murrin Murrin, which is still in operation, adopted the Sherritt process which precipitates the metals as sulphides, re-dissolution in an oxidative leach with oxygen under pressure followed by solvent extraction and hydrogen reduction. Bulong processing did not involve precipitation of the sulphide, using direct solvent extraction of the nickel and cobalt from the HPAL discharge liquor (after partial neutralisation and counter current decantation), with the metals produced by electrowinning (electrode depositing); and Cawse precipitated the nickel and cobalt as a hydroxide, originally redissolving with ammonia and refining using solvent extraction and production of nickel metal cathodes by electrowinning and precipitation of a cobalt sulphide product through addition of sodium hydrosulphide (NaHS).

The Ravensthorpe project in Western Australia uses enhanced pressure acid leach technology (similar to the Cawse Flowsheet. Similarly, the Goro project in New Caledonia utilises a process route similar to the Cawse design. While the Sumitomo's Coral Bay and Sherrit's Ambatovy projects utilise the Moa/Murrin flowsheets producing a mixed sulphide. The HPAL processing technology is no longer new, generally being considered to be in its fourth generation and is much improved by the experience gained in operating plants, as experienced by the fast commissioning and ramp up time for the Taganito Project in the Philippines (2013). There are still challenges in commissioning and ramping up HPAL operations as experienced by Highland Pacific's Ramu Operation in Papua New Guinea but these challenges are well understood.

In a scandium context, the development of HPAL for the extraction of scandium is widely accepted. Metallica Minerals completed a Pre-Feasibility study based on HPAL for scandium extraction. This included extensive metallurgical testwork. Both the Owendale (Platina), Flemington (Australian Mines) and the Nyngan (Scandium International Mining Corporation) projects have completed testwork programs and studies, including metallurgical testwork validation of HPAL for scandium extraction, with recoveries similar to that of nickel and cobalt.

The metallurgical testwork completed in the previous two feasibility studies on Syerston typically followed scandium, as well as nickel and cobalt. All historical testwork confirms that scandium extraction using HPAL ranges from 80 - 90% and higher.

### 13.1.2 Resin-in-pulp

Clean TeQ uses a proprietary ion exchange technology (Clean-iX®) for extraction and purification of metals and industrial water treatment. The development of the base technology for the Clean-iX® process was developed out of the All Russian Research Institute of Chemical Technology (ARRICT) over a period of 40 years.

The All Russian Research Institute of Chemical Technology was founded in April 1951 in Moscow, Russia, with the basic themes of research connected with the creation and development of chemical technologies for the processing of uranium and rare-metal ores and production of nuclear-pure structural materials.

ARRICT carries out a complete cycle of scientific research and development works aimed at creating profitable highly effective technologies for the production of uranium and nuclear-pure metals (lithium, beryllium, zirconium, hafnium, tantalum, niobium, etc.) for atomic industry and other branches of industry. The technologies have been adapted for processing gold-bearing, molybdenum, tungsten and other ores. ARRICT's continuous ion exchange technology has been used in several full-scale mining operations, including 22 uranium mines and 6 gold mines.

**Figure 13-1: Continuous ion exchange plant in FSU (uranium production)**

Since 1951 ARRICT has been part of the development of over 30 mining operations using the technology, mainly for uranium and gold extraction, from leached slurries and solutions.

In 2000, Clean TeQ obtained the exclusive licence for all technical information relating to ion exchange resin, ionic membranes, organic solvent extractants, including manufacturing know-how and plant design, for all countries outside the former USSR. The licence has a term of 99 years. Since that time Clean TeQ has further developed the base technology for several metal applications.

The former Director General of ARRICT's Sorption Division, Dr Nikolai Zontov is currently the Principal Scientist at Clean TeQ, bringing significant knowledge and experience in continuous ion exchange design and operation.

Since obtaining the licence, Clean TeQ has further developed the technology for base metals, uranium and gold, with particular improvements for laterite ore processing, scandium and uranium. Clean TeQ has 10 additional patents on various aspects of the technology, including one for extraction and purification of scandium.

The following is a summary of the developments completed by Clean TeQ for RIP technology:

- 2001-02: Development of RIP and desorption process for the Murrin Murrin nickel and cobalt Operation to compliment or potentially replace the Counter Current Decantation (CCD) circuit. Incorporated a large-scale plant based pilot plant.
- 2004-08: Development of RIP and desorption process for nickel and cobalt from laterite ores from HPAL leached ores – several patents lodged. Development was with BHP Billiton (BHPB) for future laterite deposits.
- 2006: Extraction and purification development of scandium from laterite ores.
- 2008: Licence signed with BHPB for nickel and cobalt technology.
- 2009: Uranium process development for alkaline and hypersaline leach solutions – patent lodged.
- 2010: Development of RIL process for gold thiosulphate leach solution.
- 2011-15: Piloting for low grade scandium recovery from acidic sulphate and chloride TiO2 process streams – patent lodged.
- 2015: Large scale piloting of scandium extraction process on Syerston ore. Lab-scale testing for nickel and cobalt on Syerston ore.

The following diagram is an overview of the Clean-iX® intellectual property developed and owned by Clean TeQ:

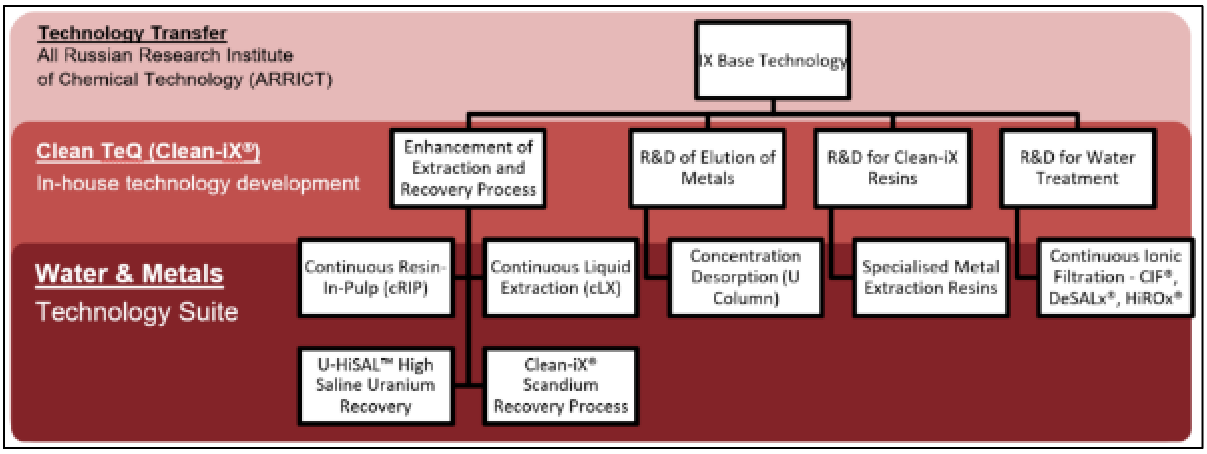



**Figure 13-2: Clean-iX® intellectual property development**

The application of RIP for scandium recovery is based on commercially applied (for other metals) and developed equipment and technologies and has the potential to offer a significant benefit for nickel, cobalt (and scandium) recovery at Syerston.

Typical nickel and cobalt recovery flowsheets use CCD's followed by precipitation and in some cases subsequent re-leaching and solvent extraction (SX) to recover the metals from re-leached slurries. This process has several disadvantages including: high capital cost due to the difficulty in separating the liquid from solids, lower recoveries due to losses in the CCD underflow, and higher capital and operating costs for re-leach and SX stages.

RIP eliminates many of these issues. RIP uses solid ion exchange resin beads directly contacted with the leached slurry to extract over 98% of the target metals contained in the solution, without the need for CCDs. Ion exchange resins are ideal for recovery and concentration of lower concentration metals, which is the case in scandium processing plants. This means that the relative chemical costs and plant size is reduced compared to SX.

## 13.2 Historical metallurgical testwork

The historical metallurgical testwork carried out both on Syerston ore and in the case of Resin-In-Pulp, other nickel laterite projects, was used as the basis for the establishment of the process design criteria for the Syerston nickel and cobalt Project PFS completed in October 2016.

Extensive metallurgical batch testwork and piloting was completed by the two previous owners, including variability testing of the different ore lithologies. This has provided a solid basis to establish the design criteria for the Project. While various metallurgical studies have been completed, the two of note completed in 2000 and 2005 are outlined below. These studies were undertaken to a feasibility level of study at the time.

During each of these testwork programs, scandium was followed in the analysis. This allowed a relatively high degree of confidence on scandium extraction using HPAL, and all subsequent unit process design criteria. However, repeat basic sighter tests were carried out by Clean TeQ to confirm the previous metallurgical test results.

### 13.2.1 Feasibility study testwork (Black Range – 2000)

Testwork for the Black Range Feasibility Study commenced in May 1999 and was completed in January 2000. The work was controlled by Black Range Minerals (previous project owners) at Lakefield Oretest (Oretest), Ammtec and Hazen laboratories, with SNC-Lavalin's involvement being on an observer basis. During this period, a program of laboratory and small-scale pilot testwork was undertaken with the aim of confirming the flowsheet selection and to provide process design criteria for the plant.

The key process units were piloted by Hazen Research Inc., Golden Colorado, USA and encompassed the following areas: High Pressure Acid Leach; counter current decantation and tailings neutralisation; Pre-reduction and neutralisation; Sulphide precipitation; Sulphide leach; Iron removal from pregnant liquor; Cobalt zinc solvent extraction; Zinc solvent extraction; and Nickel hydroxide precipitation.

Typically, the continuous runs went for 5 to 12 days, although the sulphide leach was limited to 4 to 5 hours. In addition, pilot milling of Goethite and Siliceous Goethite ores was conducted at Ammtec Laboratories in Western Australia to understand materials handling and comminution behaviours and provide slurry for piloting.

A series of batch and semi-continuous work was also carried out and included: Settling and thickening tests; Filtration tests - rate and area requirements; Tailings neutralisation; Nickel electrowinning; Limestone characterisation; High Pressure Acid Leach variability testwork; and Rheology testwork.

### 13.2.2 Feasibility study update testwork (Ivanplats Syerston – 2005)

Testwork for the Feasibility Study Update (FSU) commenced in December 2004 and was completed in February 2005. The work was commissioned by Ivanplats Syerston (previous project owners) at Oretest laboratories, with SNC-Lavalin contracted to assist with the testwork program scope definition and supervision. During this period, a program of laboratory and small-scale pilot testwork was undertaken with the aim of confirming the flowsheet selection and to provide design criteria for the plant.

The key processing units were piloted by Oretest in Perth, Western Australia. They encompassed the following areas: High Pressure Acid Leach; counter current decantation and tailings neutralisation; Pre-reduction and neutralisation; and Sulphide precipitation;

Typically, the continuous runs went for 2 to 3 days for each of the four blended composite samples. A series of batch and semi-continuous work was also carried out to support equipment selection and design parameters and included: Settling and thickening tests; Filtration tests - rate and area requirements; Limestone (stone dust) characterisation; High Pressure Acid Leach batch testwork; Rheology testwork; Agitation testwork; and Flume beaching testwork on tailings.

### 13.2.3 HPAL process design criteria values

The two metallurgical programs confirmed the following design criteria for HPAL and are well supported by testwork. Metal extractions are typical of laterite ores. Acid consumption is at the low end of the typical range as a result of the relatively low acid consumers in the feed. The conditions in Table 13-1 were assumed for the pre-feasibility study HPAL circuit.

**Table 13-1: Historical HPAL process design criteria**

| Criteria | Measurement |
|---|---|
| Leach Temperature | 250°C |
| Mean Residence Time in Leach | 70 minutes |
| Free Acid in HPAL discharge | 40 – 45 g/L |

| Criteria | Measurement |
|---|---|
| Sulphuric Acid to Leach $^{\alpha}$ | 240 kg/t |
| Estimated Leach Nickel Extractions | 97% |
| Estimated Leach Cobalt Extractions | 95.5% |
| Estimated Leach Scandium Extractions | 86% $^{\beta}$ |

Notes:
A $H_2SO_4$ consumption depends on the concentration of gangue materials. Previous feasibility testwork allowed for the establishment of a formula to calculate predicted acid consumption for a given ore composition. This is standard practice for nickel laterite projects. This same acid calculation formula was used for the 2016 PFS. β Based on a review of the historical testwork assay sheets as well as testwork completed for the August 2016 Scandium Feasibility Study.

Acid consumption is a key consideration for laterite economics, as it represents approximately 20% - 30% of the total operating cost. The main acid consuming elements are magnesium and calcium, which are common in saprolite minerals. Other acid consumers are iron and aluminium. One of the key benefits of using a HPAL process is that the majority of the acid used to leach iron is "returned" through the process, giving much lower acid consumption than typical atmospheric and heap leach operations.

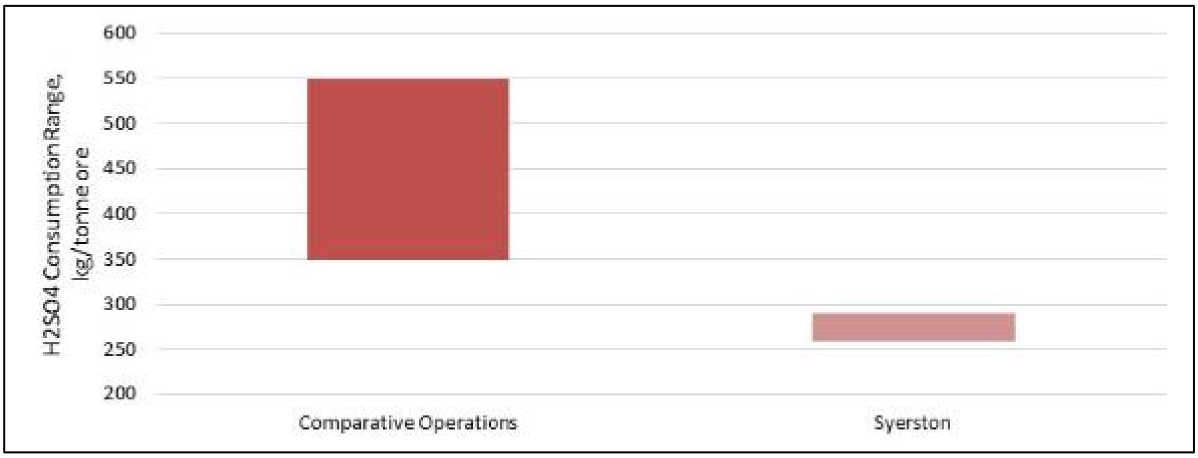



**Figure 13-3: Indicative comparative acid consumption of Syerston and other Laterite operations**

Most laterite operations have acid consumption values of 350-500 kg $H_2SO_4$/t of ore treated. Syerston's mineralogy, being predominately goethite, high iron, with little acid consuming materials, low magnesium and calcium, has acid consumption typically ranging between 240-300 kg/t. This comparatively low acid consumption provides a significant cost advantage for Syerston. Operating at 100 - 150 kg/t below typical laterites represents a 10-15% reduction in operating cost for Syerston.

### 13.2.4 Clean TeQ RIP historical testwork

Clean TeQ has undertaken nickel, cobalt and scandium RIP extraction testwork on Syerston and other project's and operation's laterite ores. From 2004-2008, Clean TeQ undertook a development program with BHP Billiton for the application of RIP technology for nickel and cobalt extraction and purification from laterite ores. During this time, laboratory scale, bench scale and large fully automated pilot programs were undertaken to develop the design criteria of the system on a HPAL pulp to develop extensive process design criteria for the complete system.

In 2014-2015, Clean TeQ also undertook extensive laboratory scale and large pilot scale testing on HPAL and RIP recovery of scandium from Syerston ore. This testwork also followed nickel and cobalt (however, both were in relatively minor concentration). Testwork included multiple leaching, resin extraction and elution tests on a range of ores to establish the design criteria for the process.

The piloting campaign ran for three consecutive weeks and processed over 12 tonnes of Syerston ore. Subsequent to this pilot campaign, resin cycling, elution and HPAL variability work was carried out targeting optimised scandium recovery.


**Figure 13-4: Overview of laterite scandium extraction RIP pilot plant**

Clean TeQ has also undertaken scandium RIP extraction testwork from other project's titanium ores. Clean TeQ has been developing a scandium extraction process for titanium dioxide waste streams for the last six years. While there are some differences in the composition of the leach solution, the overall extraction and desorption chemistry are similar.

### 13.2.5 Clean TeQ Post RIP testwork

The back end 'refining' aspects of the processing flowsheet, i.e. post RIP, is a new addition to the overall flowsheet and is sufficiently different to warrant additional testwork. Sighter level downstream testwork was undertaken by Clean TeQ at ALS Ammtec on the proposed flowsheet. Synthetic solution was prepared to a specification designed to simulate an intermediate product discharging the scandium circuit and expected for a nickel/cobalt recovery circuit proposed for the Syerston Project. The testing included:

- Synthetic solution preparation
- Impurity removal:
  - Acid neutralisation
  - Iron removal
  - Sulphide precipitation
  - Solvent extraction
- Cobalt solvent extraction
- Nickel Solvent extraction.

These process units reflect those adopted for the Syerston Ni/Co Project, using standard solvent extractants and typical industry conditions and as a result, this work demonstrated impurity removal extents as well as SX loading, scrubbing and stripping extents under specific conditions. These tests were undertaken to support the Prefeasibility Study using a synthetic liquor representative of those ores. As a result, it is not completely representative nor extensive enough to support the detailed design of the Syerston Ni/Co Project and is not optimised for the project. More testwork is planned for this aspect of the plant as well as the downstream crystallisation processes which are yet to be tested. Design parameters are supported by industry practice for nickel and cobalt solvent extraction projects and operations which are well established.

### 13.2.6 Future testwork

The following testwork has been identified by Clean TeQ as being required for subsequent test programs to optimise the process routes and further reduce any technical risk:

- HPAL:
  - No further work is required due to the extensive amount of work completed over the three feasibility studies (two nickel/cobalt and one scandium) carried out on the project to date.
  - Additional bulk HPAL tests are required to provide feed for the planned downstream RIP and Nickel/Cobalt Purification and crystallisation testing. This will provide an opportunity to further understand and optimise the HPAL process on Syerston ores.
- RIP:
  - Demonstrate sustained RIP performance and are in line with the PDC values used in the PFS. There is an expectation that the testwork can continue to be improved.
  - Confirmatory sequential resin loading tests to be carried out to simulate the sequential loading of scandium, followed by nickel/cobalt using RIP.
  - Sufficient loaded nickel/cobalt resin should be generated to allow for a large sample of nickel sulphate and cobalt sulphate to be produced.
- Nickel/Cobalt Purification & Crystallisation, specifically solvent extraction of RIP products:
  - There is limited SX testwork on the RIP products. This requires improved understanding moving into the FS level of plant design.
  - Development and optimisation testwork should be carried out using nickel/cobalt loaded resin eluate to determine the final process design criteria for the production of sulphate products, thereby testing the nickel and cobalt sulphate crystallisation and drying steps, demonstrating the final product specification.
  - This will also allow for samples to be generated for prospective customers.
- Scandium Purification:
  - No further testwork is required as this was extensively tested in the Scandium Feasibility Study and is not considered in the nickel and cobalt Project base case.

# 14 Mineral Resource Estimates

Section 14 is presented in two parts, Section 14.1 relates to Resource Estimates carried out by McDonald Speijers in 2005, updated in 2016 and verified by SRK in 2017. This was used as the basis for the Mineral Reserve being reported in this Technical Report.

Section 14.2 relates to a Resource Estimate carried out by Widenbar in 2017. Section 14.2 is the Resource Estimate being reported/ released as part of this Technical report.

## 14.1 Mineral Resource estimates 2005 to 2016

### 14.1.1 Accepted data

In 2005 McDonald Speijers accepted a total of 1,228 holes (1,183 RC holes and 45 selected aircore holes) for use in resource estimation. In 2016 this increased to 1,318 RC holes and 45 aircore holes for a total of 1,363 holes. The additional data represented only 8% by number of holes and 6% by length of total accepted drill hole data.

RAB holes were rejected en masse because of potential contamination and lack of confidence in collar coordinates.

Aircore holes were subject to doubt because:

- Collar coordinates could not be properly validated
- There were indications of a possible grade bias between aircore and RC holes
- Many failed to fully penetrate the laterite profile.

They were only accepted in areas where there was no effective coverage by the RC drilling pattern. Ultimately, aircore holes made up only 3% by length of the accepted drill hole data.

The groups of multiple RC twin holes drilled around Calweld holes in 2005 were excluded because of their extreme clustering.

Calweld holes were also excluded because of their extreme difference in sample support and because they tended not to penetrate the complete laterite profile. With only one exception (in a poorly mineralised location) they were surrounded by systematic patterns of RC holes that provided adequate data.

However, it should be noted that all existing drill holes were taken into account during interpretation of laterite zones and the dunite boundary.

### 14.1.2 Statistical data analysis

#### Sample statistics

Statistics in the following tables are based on all accepted drill holes. Significant clustering effects may exist due to variations in the drilling pattern (Figure 14-1). No declustering procedures have been applied.

Summary statistics for raw samples and for 2 m composites in the accepted drill holes are shown in Table 14-1 and Table 14-2.

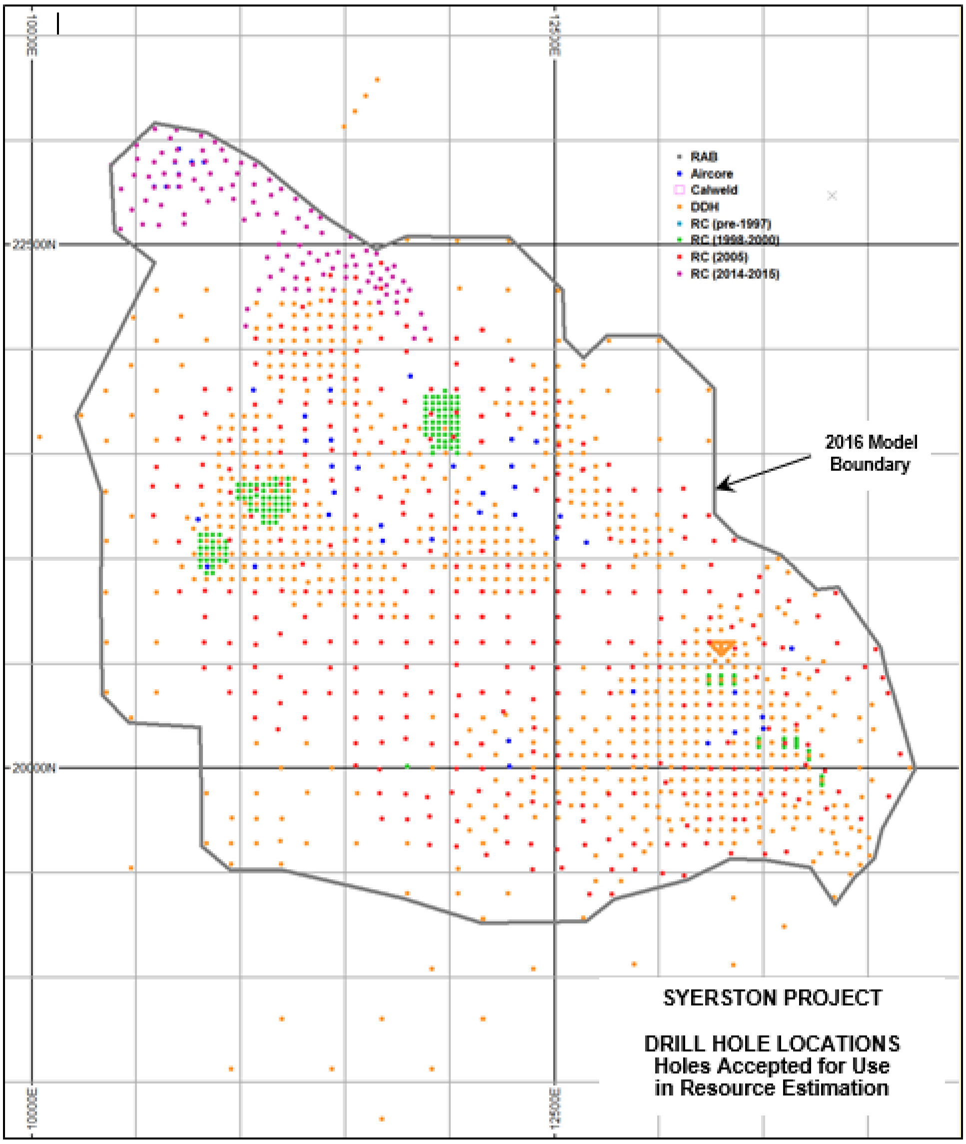



**Figure 14-1: Drill hole layout accepted holes plan view**

**Table 14-1: Summary sample statistics, accepted drill holes (after merging assay & geology files)**

| File | Field | Number of records | Number of values | Minimum | Maximum | Mean | Standard deviation |
|---|---|---|---|---|---|---|---|
| Syds | Ni% | 42337 | 41048 | 0.0005 | 4.19 | 0.41 | 0.38 |
| Syds | Co% | 42337 | 41048 | 0.00025 | 2.72 | 0.064 | 0.104 |

## Assay population characteristics and zone codes

In 2005, a brief review of sample assay population characteristics was conducted to aid in the selection of criteria to support interpretation of laterite zones. This was based on 1 m RC samples below the interpreted base of alluvium. Details were presented in our 2005 report.

That initial statistical analysis and subsequent directional variography revealed distinct differences in assay population characteristics and spatial grade distribution patterns for Co and Ni in the GZ and SGZ between the north-western and south-eastern parts of the deposit. Consequently, the deposit was divided into two sub-domains along the approximate centre line of the main palaeochannel, as shown in Figure 14-2.

Primary zone codes (block model code ZONE) were refined by adding a supplementary zone code (block model code IZONE). This was used to select datasets for further statistical analysis and variography and to control subsequent grade interpolation into the resource model.

**Table 14-2: Summary composite statistics, accepted drill holes (after application of top-cuts)**

| File | Field | Number of records | Number of values | Minimum | Maximum | Mean | Standard deviation |
|---|---|---|---|---|---|---|---|
| Syzdt | Ni% | 28536 | 27404 | 0.001 | 2.50 | 0.405 | 0.364 |
| Syzdt | Co% | 28536 | 27404 | 0.000 | 1.000 | 0.062 | 0.088 |

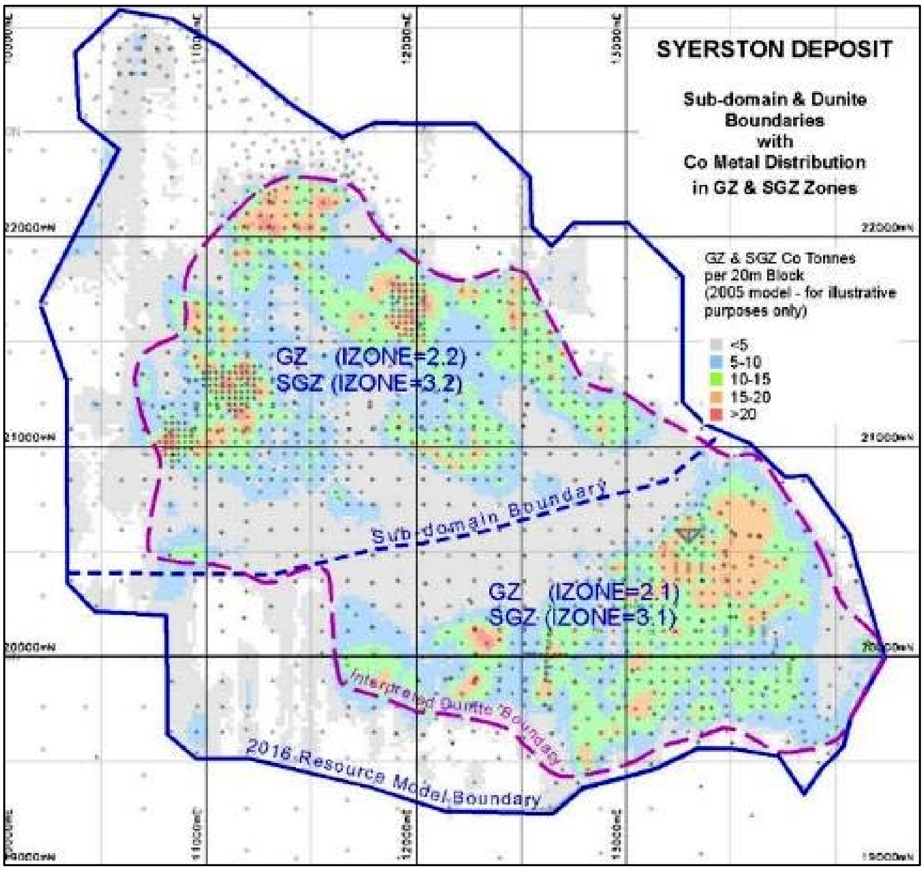



**Figure 14-2: Sub-domain boundary applied to GZ & SGZ**

## Assay top-cuts

In 2005, a range of tests were conducted to assist in selection of assay top-cuts. They were:

- Examination of cumulative log-probability plots for any distinct inflections at the high grade extremities.
- Examination of normal and log histograms to identify levels at which it became evident that isolated values lay outside the limits of the bulk of the distributions.
- Iterative tests to determine the cut required to bring the lognormal mean estimator into line with the arithmetic mean, after progressively removing low outliers until the variance stabilised.
- Empirical tests in areas of 30 m infill drilling carried out in early 2005 to establish the approximate top-cuts required to bring pre-infill average grade estimates into line with post-infill estimates for the volumes of some stage-1 pit designs from the 2000 feasibility study. This indicated that in order to reduce risks associated with unusually high Co grades in holes outside areas of 30 m infill

(60-120 m hole spacing), a harsher Co top-cut would be prudent.

- Examination of plans and sections to confirm that values above the selected thresholds did not form coherent, discrete, high-grade zones.
- The top-cuts selected in 2005 were accepted unchanged for use in an updated resource estimate in 2016. They are summarised in Table 14-3; refer to SRK's 2005 report for greater detail.

**Table 14-3: Assay top-cuts**

| Geology Zone | IZONE | 2016 Estimate | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | Ni % | Co% | | Pt g/t | Mg % | Mn % | Cu % |
| | | | 30 m Pattern | 60 m+ Pattern | | | | |
| AV | 11 | - | - | - | 1.2 | - | - | - |
| OVB | 12 | - | - | - | 1.2 | - | - | - |
| TZ | 1 | 1.5 | 0.15 | 0.15 | 1.2 | 7.5 | 2.0 | 0.055 |
| GZ | 2.1/2.2 | 2.5 | 1.0 | 0.5 | 1.2 | 6.0 | - | 0.180 |
| SGZ | 3.1/3.2 | 2.25 | 0.35 | 0.25 | 1.2 | 9.0 | - | 0.100 |
| SAP | 4 | 1.75 | 0.1 | 0.1 | 0.8 | - | - | 0.050 |

In early 2005, three areas covering stage-1 pit designs from the 2000 feasibility study were infill drilled by RC on a 30 m pattern. This was done to test the impact on average Co grades within the proposed pit volumes, which had focused on high Co values in earlier, wider spaced drilling. Our experience with other, similar deposits had shown this to be an area of high potential risk.

A number of comparative block model estimates were generated for these test areas, with and without the 30 m infill data. These indicated that the infill drilling resulted in a 15%-20% drop in contained Co metal, while contained Ni remained virtually unchanged.

The results of the exercise strongly suggested that a harsher Co top-cut would be prudent in areas drilled at wide spacings (60 m or more) to reduce the risk of serious overestimation of Co grade within pit designs used during the early stages of any Ni-Co project.

Iterative modelling in the test areas indicated that, when the 30 m infill drilling was excluded, the selected Co top-cuts of 1% in the GZ and 0.35% in the SGZ had to be reduced to about 0.5% and 0.25% respectively to give approximately the same average cobalt grade in the Stage-1 pit volumes. No changes were required for nickel top-cuts.

This might result in some limited degree of underestimation of global average cobalt grade, but McDonald Speijers felt that it was more important to ameliorate risk associated with the crucial early stages of any future Ni-Co operation.

## Variography

In 2005, an extensive three-dimensional variographic study was conducted using Datamine software. This involved the 2 principal elements (nickel and cobalt) and six accessory elements nominated at the time by Ivanplats as being of the most potential significance (Pt, Fe, Si, Mg, Al and Mn).

Directional variograms were generated for each of the six geological domains using 2 m drill hole composites, with the GZ and SGZ subdivided into northwest and southeast sub-domains as previously described. Details were reported in our 2005 report.

Because the proportion of additional drill hole data was very small and the time required would have been disproportionately large, the variographic study was not repeated. The variogram model parameters used in 2005 were accepted without change.

They were generally two or three structure models based on normalised log variograms for Nickel and cobalt (Figure 14-3 and Figure 14-4).).

Variogram model parameters are listed in Table 14-5.

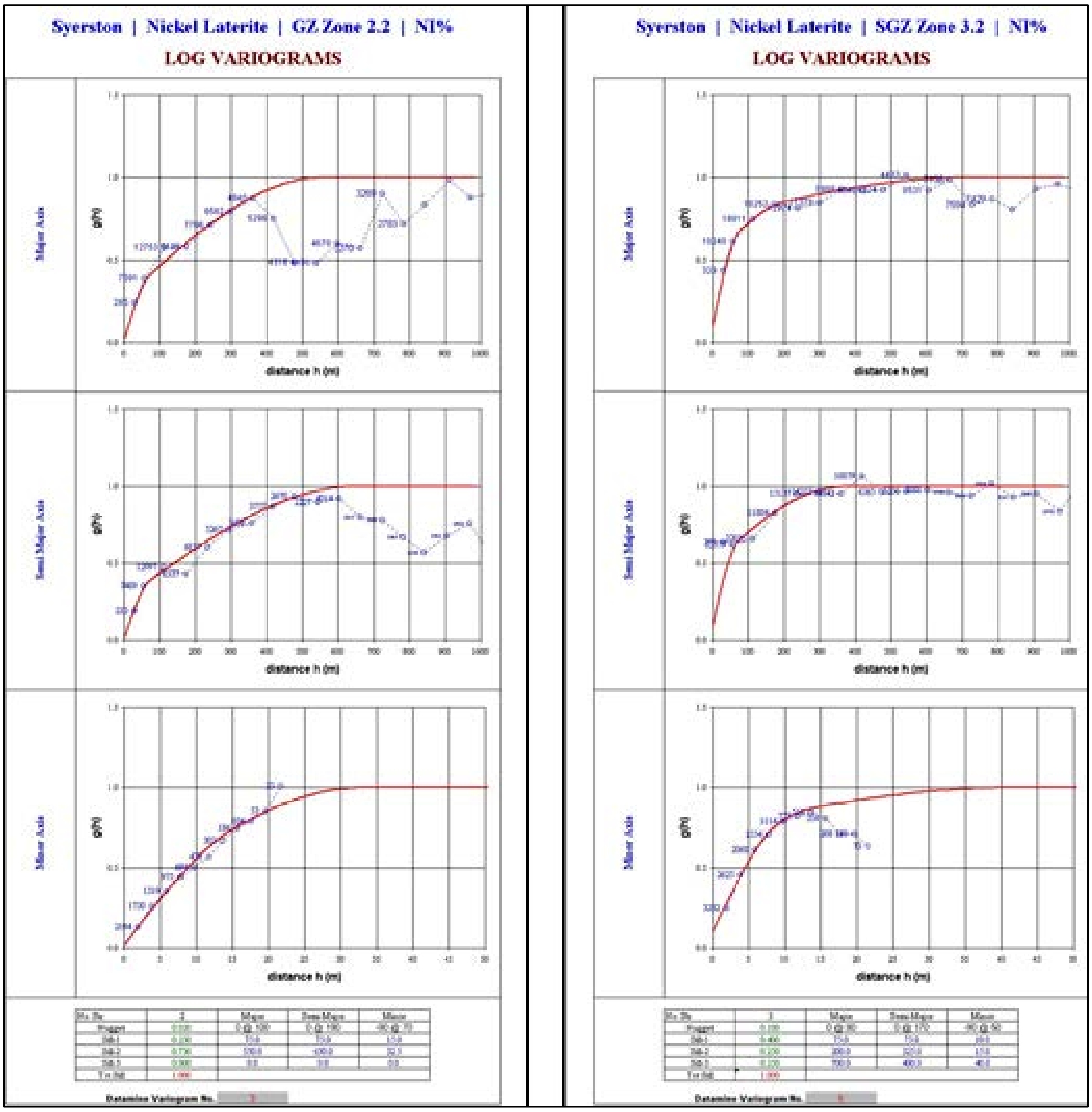



**Figure 14-3: Ni variograms for GZ and SGZ**

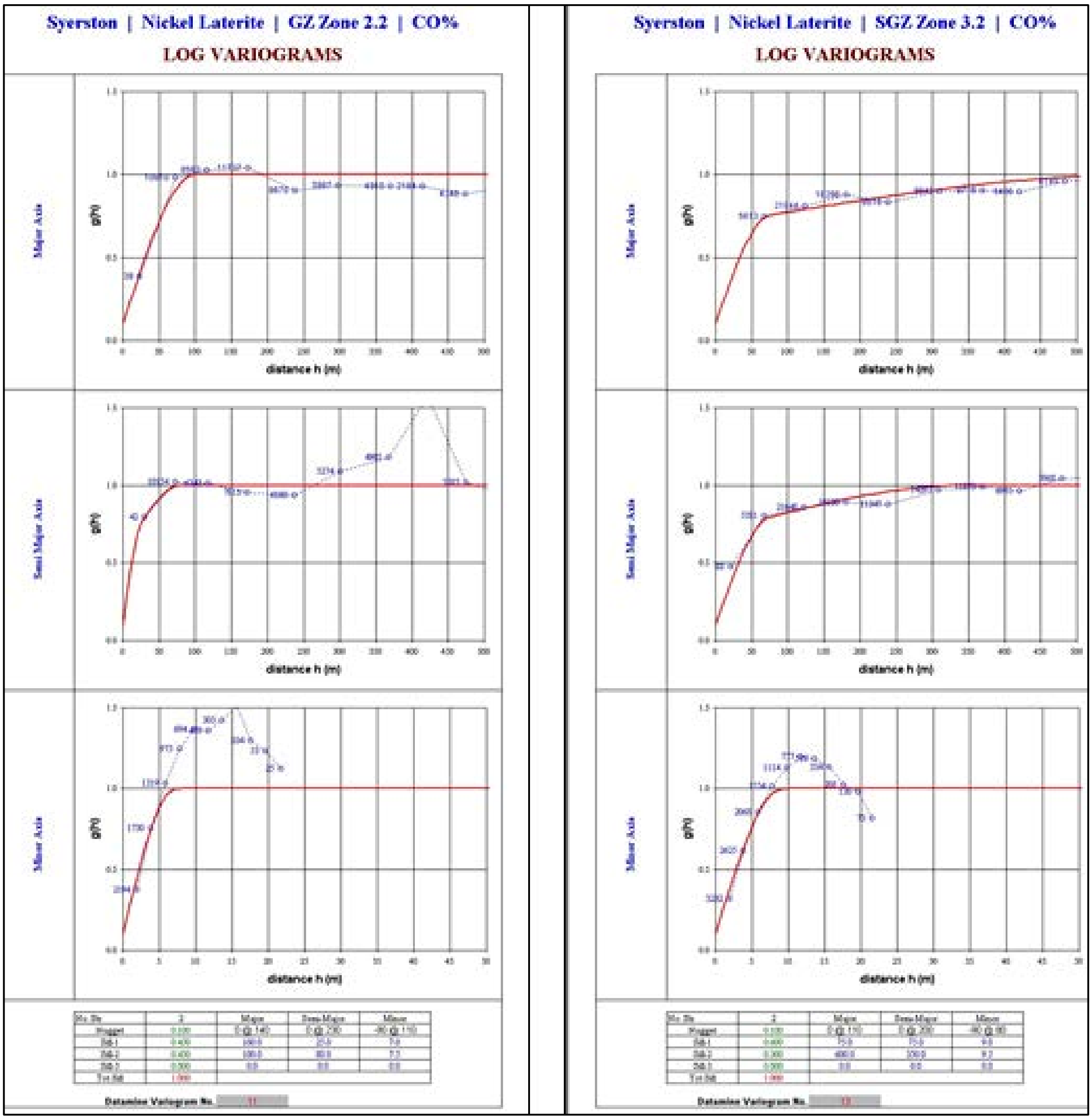



**Figure 14-4: Co variograms for GZ and SGZ**

### 14.1.3 Resource estimation procedures

A three-dimensional resource block model was generated using Datamine software, with block grades estimated for nickel and cobalt, and also for the accessory elements Pt, Fe, Si, Al, Ca, Cu, Cr, Mg, Mn, and Zn. Estimates for Ni, Co, Pt, Fe, Si, Mg, Al and Mn were made by ordinary kriging. Other elements were estimated using an anisotropic inverse distance squared interpolation method.

#### Outline of procedure

Laterite zones boundaries interpreted as strings on north–south section lines in 2005 were updated to take account of subsequent drilling results.

These strings were linked to form surface wireframes (DTM's) representing:

- Top of saprolite
- Top of SGZ
- Top of GZ

- Top of TZ
- Base of Alluvium.

Wireframes were filled above or below with blocks and combined in a controlled sequence to generate a 3D block model. The model was confined within a plan perimeter defining the limits of the area covered by accepted drill holes (Table 14-4).

Horizontal block dimensions of 20 x 20 m were selected in relation to typical drill hole spacings. The block height was based on advice about a likely open pit working bench height. Model limits were extended to cover additional drilling in the northern part of the deposit.

**Table 14-4: Block model physical parameters**

| Dimension | East–west (X) | North–south (Y) | Vertical (Z) |
|---|---|---|---|
| Co-ordinate limits | 9970 - 14370 | 18850 – 23630 | 220-320 |
| Cell size (m) | 20 | 20 | 2 |
| Number of cells | 220 | 234 | 50 |

Blocks intersected by the topographic surface were sub-celled to 0.5 m in the vertical direction

Model blocks and drill hole samples were flagged with zone codes (Table 14-4).

The interpreted dunite boundary string was used to flag model blocks with a code indicating interpreted bedrock type. The code was DUNITE=1 (dunite) or DUNITE=0 (pyroxenite).

Top-cuts were applied to samples which were then composited to 2 m within each laterite zone, with a 1 m minimum composite.

Block grades were interpolated under zonal control. Because the zone boundaries were expected to be gradational in nature and potentially quite irregular, a degree of controlled transparency was allowed across all zone boundaries between the base of overburden and the top of the saprolite, to emulate some mixing of material types during mining.

Data search ellipsoids were defined independently of variogram models, based on either the first or second variogram structure depending on whether they were two or three component models. Horizontal ranges were scaled so that the minimum horizontal search distance in any direction was 100 m (the minimum required to find at least the nearest hole in all directions for the predominant 60 x 60 m drilling pattern). Grade estimation parameters are summarized in Table 14-6.

Block grades were estimated by ordinary kriging or anisotropic inverse distance squared interpolation as specified previously.

**Table 14-5: Variogram model parameters**

| Element | IZONE | VREFNUM | Major axis | | Nugget | Structure 1 | | | | Structure 2 | | | | Structure 3 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Azimuth | Dip | | Semi-major axis | Major axis | Minor axis | Spatial variance | Semi-major axis | Major axis | Minor axis | Spatial variance | Semi-major axis | Major axis | Minor axis | Spatial variance |
| Ni | 1 | 1 | 60 | 0 | 0.1 | 50 | 150 | 8 | 0.5 | 400 | 300 | 8.5 | 0.4 | 0 | 0 | 0 | 0 |
| Ni | 2.1 | 2 | 95 | 0 | 0.02 | 75 | 125 | 10 | 0.175 | 1500 | 1250 | 40 | 0.805 | 0 | 0 | 0 | 0 |
| Ni | 2.2 | 3 | 100 | 0 | 0.02 | 75 | 75 | 15 | 0.25 | 650 | 550 | 32.5 | 0.73 | 0 | 0 | 0 | 0 |
| Ni | 3.1 | 4 | 90 | 0 | 0.1 | 75 | 100 | 15 | 0.3 | 900 | 900 | 27.5 | 0.6 | 0 | 0 | 0 | 0 |
| Ni | 3.2 | 5 | 80 | 0 | 0.1 | 75 | 75 | 10 | 0.4 | 325 | 200 | 15 | 0.25 | 400 | 700 | 40 | 0.25 |
| Ni | 4 | 6 | 70 | 0 | 0.02 | 75 | 125 | 10 | 0.2 | 500 | 600 | 100 | 0.15 | 3000 | 1500 | 200 | 0.63 |
| Ni | 11 | 7 | 70 | 0 | 0.02 | 100 | 75 | 10 | 0.3 | 500 | 600 | 75 | 0.23 | 800 | 1750 | 100 | 0.45 |
| Ni | 12 | 8 | 135 | 0 | 0.02 | 100 | 125 | 10 | 0.35 | 750 | 300 | 12 | 0.2 | 1000 | 1500 | 12 | 0.43 |
| Co | 1 | 9 | 140 | 0 | 0.1 | 100 | 100 | 12 | 0.45 | 400 | 400 | 15 | 0.45 | 0 | 0 | 0 | 0 |
| Co | 2.1 | 10 | 50 | 0 | 0.1 | 25 | 25 | 7.5 | 0.2 | 100 | 125 | 8 | 0.5 | 350 | 750 | 8.5 | 0.2 |
| Co | 2.2 | 11 | 140 | 0 | 0.1 | 25 | 100 | 7 | 0.45 | 80 | 100 | 7.5 | 0.45 | 0 | 0 | 0 | 0 |
| Co | 3.1 | 12 | 30 | 0 | 0.1 | 75 | 75 | 10 | 0.35 | 300 | 600 | 10.5 | 0.55 | 0 | 0 | 0 | 0 |
| Co | 3.2 | 13 | 110 | 0 | 0.1 | 75 | 75 | 9 | 0.6 | 350 | 600 | 9.5 | 0.3 | 0 | 0 | 0 | 0 |
| Co | 4 | 14 | 120 | 0 | 0.05 | 75 | 75 | 7 | 0.2 | 250 | 300 | 8 | 0.15 | 3000 | 3000 | 40 | 0.6 |
| Co | 11 | 15 | 90 | 0 | 0.05 | 50 | 50 | 15 | 0.2 | 175 | 200 | 25 | 0.15 | 500 | 1200 | 30 | 0.6 |
| Co | 12 | 16 | 95 | 0 | 0.05 | 50 | 50 | 7.5 | 0.35 | 175 | 100 | 12 | 0.2 | 500 | 1000 | 12.5 | 0.4 |

**Table 14-6: Interpolation parameters**

| | | |
|---|---|---|
| Interpolation Method | Ordinary Kriging | Ni, Co, Pt, Fe, Si, Mg, Al, Mn |
| | Inverse distance squared | Cr, Ca, Cu, Zn |
| Search Parameters | By IZONE | |
| Octant Search | Not used | |
| Minimum & Maximum Composites | Pass 1 | Minimum: 8, Maximum: 24 |
| | Pass 2 | Minimum: 4, Maximum: 24 |
| | Pass 3 | Minimum: 2, Maximum: 16 |
| Search Radii Multipliers | Pass = 1, Pass 2 = 2, Pass 3 = 4 | |

## Model validation

Checks by McDonald Speijers included:

- Extensive visual comparisons between drillholes grades and block grades, in section and plan views
- Comparing average composite grades with average block grades for series of regular slices through the model in section and in plan (swath plots). Due to the large local variations in drill hole spacings, it was necessary to first decluster the composite values.

SRK completed addition statistical comparison and swath plot checks using declustered composites on a zone by zone basis and found good agreement between the model and the composites. Examples for the main mineralised GZ are shown in Figure 14-5 and Figure 14-6.

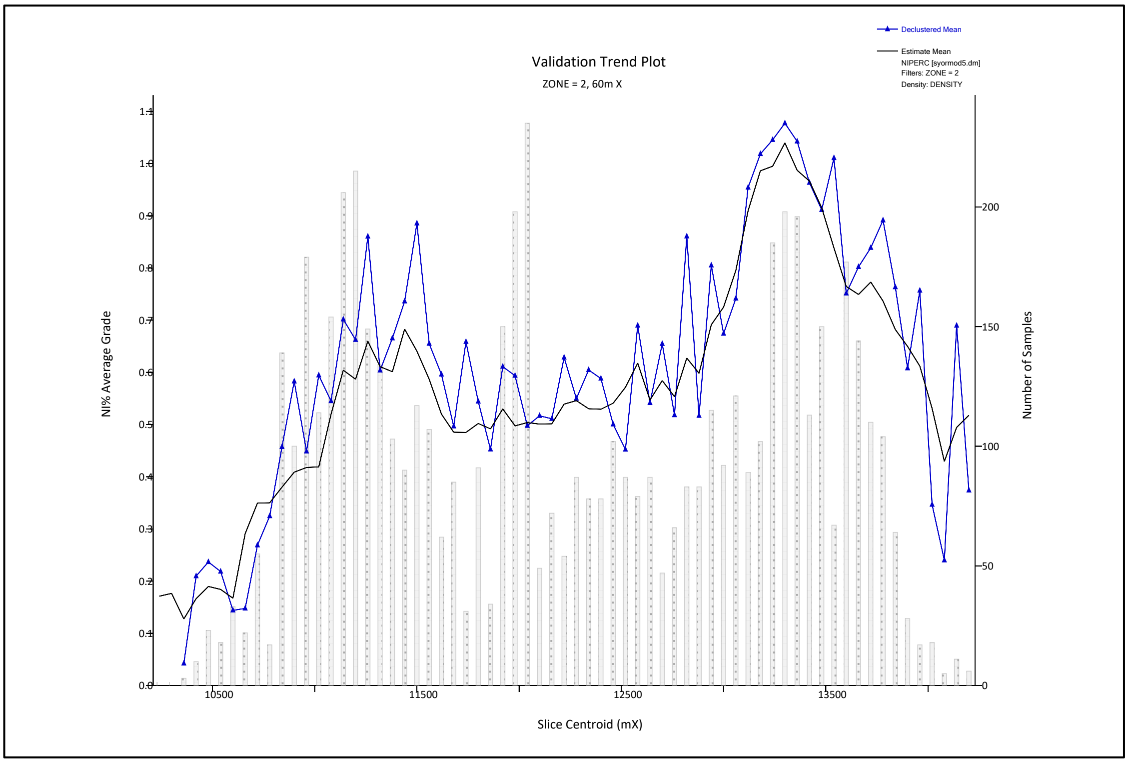



**Figure 14-5: GZ (Zone 2) Easting swath plot for Ni**

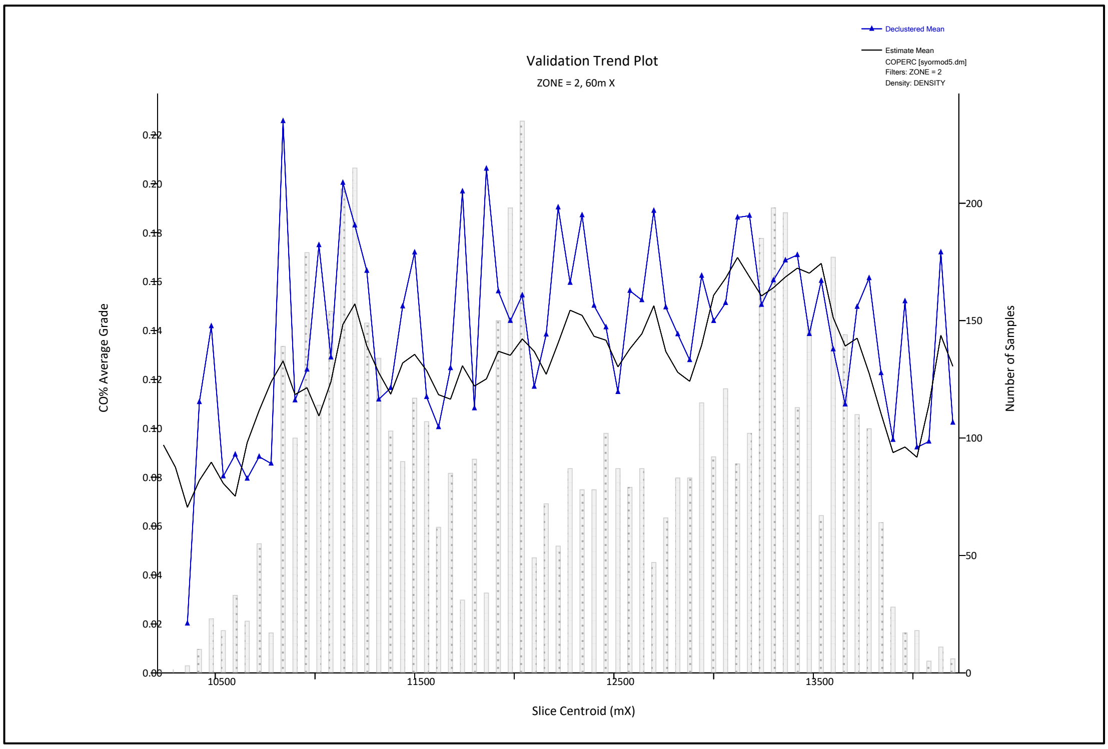



**Figure 14-6: GZ (Zone 2) Easting swath plot for Cobalt**

SRK also created grade shells using Leapfrog and compared the NiEq 0.6% shell to the shell produced by the McDonald Speijers block model at 0.6% NiEq. The shells compared very well. An example section is shown in Figure 14-7.

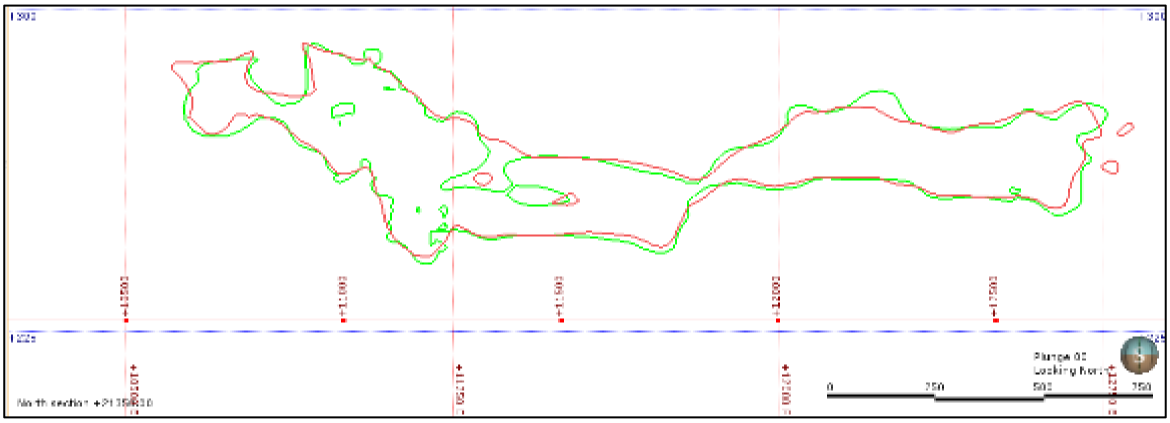


**Figure 14-7: Block model (red) and leapfrog (Green) 0.6% NiEq shells section 21350N**

## Resource classification

The resource model was classified using the terminology and guidelines of the Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code) which is effectively equivalent to the CIM definition standards 2014. The criteria were developed principally on the basis of:

- An assessment of overall sampling and assaying reliability
- A geological assessment of levels of confidence in the continuity and geometry of the main mineralised zones provided by various drill hole patterns
- A 2005 review of average kriging variances, kriging efficiencies and kriging slopes of regression for both nickel and cobalt block grade estimates in the two main mineralised zones (GZ & SGZ).

The outcome was a classification related primarily to drilling patterns:

- Measured: Consistent 60 x 60 m pattern or closer
- Indicated: Consistent 120 x 120 m pattern or closer
- Inferred: Other areas within the model boundary.

Classifications were applied to the block model in a vertical sense using plan strings as shown in Figure 14-8. The proportion of the estimated resource classified as Measured in 2005, and again in 2016, was considerably lower than it had been in 2000 (about 50% compared with over 80% previously).

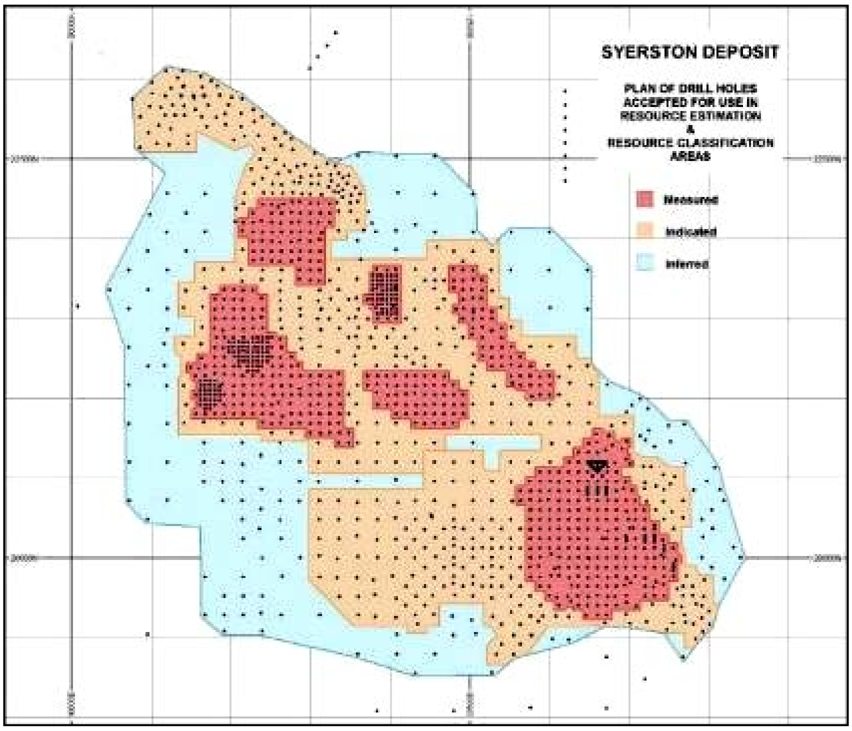



**Figure 14-8: Resource classification boundaries**

Although some 2015 drilling in areas north of about 22,240 mN resulted in patterns approximating 60 x 60 m, no material in this area was classified as Measured because of inadequate sampling and assaying quality control procedures for nickel and cobalt and because McDonald Speijers was unable to adequately verify sampling procedures.

## Cut-off grade & nickel equivalent factors

In May 2005, Ivanplats specified:

- Assumed metal prices for nickel and cobalt
- Expected metallurgical recoveries for nickel and cobalt from the beneficiation circuit and from autoclave feed
- Estimated revenue factors allowing for anticipated post mine gate product treatment and refining charges
- Estimated unit operating costs per tonne of feed.

In 2016, Scandiun21 reviewed these and advised that they remained appropriate (Figure 14-8). In 2005, McDonald Speijers used those parameters to estimate net dollar values per 1% per tonne for nickel and cobalt and derived an NiEq factor for Cobalt. The resulting nickel equivalent grade formula was: NiEq% = Ni% + (Co% * 2.95). This remains unchanged.

## Table 14-7: NiEq cut-off grade assumptions

| | | | | | | |
|---|---|---|---|---|---|---|
| **Assumptions - Revenue & Costs** | | | | | | |
| Exchange rate US$:A$ | 0.70 | | | | | |
| Metal prices: | | | | | | |
| Nickel US$/lb | 4.00 | | | | | |
| Cobalt US$/lb | 12.00 | | | | | |
| Gross revenue as % of metal price (covering product treatment & refining costs) | 85.0% | | | | | |
| *Comment: Revenue factor assumed to be the same for Ni & Co. Not usually so.* | | | | | | |
| Operating cost average years 3 – 20: | | | | | | |
| All up A$ / t **autoclave feed** [incl transport & royalty] | 66.71 | | | | | |
| Excluding mining A$ / t **autoclave feed** [incl transport & royalty] | 60.24 | | | | | |
| Autoclave feed rate tpa | 2,500,000 | | | | | |
| **Assumptions - Resource** | | | | | | |
| Estimated proportions of contained metal in resource by Si content: | | | | | | |
| (based on old NIEQ2000 COG of 1.65%) | | | | | | |
| **Si Ranges** | **All** | **<15%** | **15-23%** | **>23%** | | |
| Mt | 104.49 | 59.65 | 29.77 | 15.07 | | |
| Proportion of Tonnes | 100.0% | 57% | 28% | 14% | | |
| Ni% | 0.66 | 0.68 | 0.67 | 0.57 | | |
| Ni_tonnes | 687288 | 403362 | 198465 | 85461 | | |
| Proportion of Ni Tonnes | 100% | 59% | 29% | 12% | | |
| Co% | 0.107 | 0.127 | 0.087 | 0.069 | | |
| Co_tonnes | 112183 | 75787 | 25957 | 10439 | | |
| Proportion of Co Tonnes | 100% | 68% | 23% | 9% | | |
| **Assumptions - Recoveries** | | | | | | |
| Beneficiation circuit recoveries to autoclave feed: | | | | | | |
| Mass | 91.0% | 100.0% | 82.0% | 73.0% | | |
| Nickel & Cobalt | | 100.0% | 94.0% | 93.0% | | |
| Metal recovery from autoclave feed to mixed sulphide (final mine product); | | | | | | |
| Nickel | 92.4% | | | | | |
| Cobalt | 90.8% | | | | | |
| **Calculations - Recovery to Product** | | | | | | |
| **Si Ranges** | **All** | **<15%** | **15-23%** | **>23%** | | |
| Overall recoveries from beneficiation circuit feed to final product: | | | | | | |
| Nickel | | 92.4% | 86.9% | 85.9% | | |
| Cobalt | | 90.8% | 85.4% | 84.4% | | |
| Weighted average recoveries from beneficiation circuit feed to final product: | | | | | | |
| Nickel | 90.0% | | | | | |
| Cobalt | 88.9% | | | | | |
| *Comments:* | | | | | | |
| *No grade-recovery curves available.* | | | | | | |
| *Recoveries assumed to be constant at all grades.* | | | | | | |
| **Net Dollar Values / Tonne** | | | | | | |
| **Si Ranges** | **All** | **<15%** | **15-23%** | **>23%** | | |
| **Nickel** | | | | | | |
| Assumed Ni% grade | 0.66 | 0.68 | 0.67 | 0.57 | | |
| Contained Ni (lb/t) | 14.501 | 14.991 | 14.771 | 12.566 | | |
| Gross In Situ Value ($US) | 58.01 | 59.97 | 59.08 | 50.27 | | |
| Gross In Situ Value ($A) | 82.86 | 85.67 | 84.41 | 71.81 | | |
| Recovered Ni (lb) | 13.050 | 13.852 | 12.829 | 10.799 | | |
| Recovered Ni gross value (A$) | 74.57 | 79.15 | 73.31 | 61.71 | | |
| Payable Ni value ($A) | 63.39 | 67.28 | 62.31 | 52.45 | | |
| Payable Ni value ($A) / 1% / tonne | 96.37 | 98.94 | 93.01 | 92.02 | | |
| **Cobalt** | | | | | | |
| Assumed Co% grade | 0.107 | 0.127 | 0.087 | 0.069 | | |
| Contained Co (lb/t) | 2.367 | 2.800 | 1.918 | 1.521 | | |
| Gross In Situ Value ($US) | 28.40 | 33.60 | 23.02 | 18.25 | | |
| Gross In Situ Value ($A) | 40.58 | 48.00 | 32.88 | 26.08 | | |
| Recovered Co (lb) | 2.105 | 2.542 | 1.637 | 1.285 | | |
| Recovered Co gross value (A$) | 36.09 | 43.58 | 28.06 | 22.02 | | |
| Payable Co value ($A) | 30.68 | 37.04 | 23.85 | 18.72 | | |
| Payable Co value ($A) / 1% / tonne | 285.74 | 291.69 | 274.19 | 271.27 | | |
| **Metal Equivalence** | | | | | | |
| % Ni equivalent to 1% Co | 2.97 | 2.95 | 2.95 | 2.95 | | |
| **Average Cutoff Grade (Operating Breakeven Basis)** | | | | | | |
| **Si Ranges** | **All** | **<15%** | **15-23%** | **>23%** | | |
| Tonnes Mined per tonne autoclave feed | 1.10 | 1.00 | 1.22 | 1.37 | | |
| Overall Operating Breakeven Cutoff (EqNi%) | 0.69 | 0.67 | 0.72 | 0.72 | | |
| SAY | 0.70 | 0.70 | 0.70 | 0.70 | | |
| Mill Feed Breakeven Cutoff (EqNi%) | 0.63 | 0.61 | 0.65 | 0.65 | | |
| SAY | 0.60 | 0.60 | 0.60 | 0.70 | | |
| **Approximate Relationship to Old NIEQ2000 Values** | | | | | | |
| Based on Ni equivalent factor for individual Si classes | | | | | | |
| Old NIEQ% (NIEQ2000) | 1.05 | 0.65 | 0.75 | 0.67 | 0.75 | 0.65 |
| New NIEQ% | 0.97 | 0.60 | 0.69 | 0.63 | 0.70 | 0.60 |

Source: 160819 Syerston Ni-Co Project - Updated Mineral Resource Estimate.pdf ; Table 9.11

## Mining assumptions

The deposit is amenable to conventional open pit mining and two feasibility studies have developed practicable staged open pit mine plans.

The most recent feasibility study was based on conventional open pit mining by contractor, using large backhoes and trucks, operating on working benches 2 m in height. Open pit production rates peaking at about eight million tonnes per annum (Mtpa) to stockpiles were envisaged, providing about 2.5 Mtpa of feed to a processing plant.

## Metallurgical assumptions

A substantial amount of metallurgical testwork was undertaken as part of the feasibility studies conducted in 2000, 2005 and 2016.

In 2005 McDonald Speijers examined the locations of 99 composites from RC drill holes that were used for metallurgical variability testwork. McDonald Speijers concluded that from a geological and resource perspective they should have provided a reasonable representation of material likely to be mined. Their average nickel and cobalt grades were higher than average resource grades, particularly for cobalt, but within the likely range of production grades, particularly for the early years of a project.

A total of 16 larger composites prepared from five of the large diameter Calweld holes were used for comminution and materials handling tests. These appeared to be located in typical mineralised material, but the small number of holes means that the composites might not necessarily provide an accurate representation of average deposit properties.

Sufficient work has been done to demonstrate that a potentially viable treatment process is available for the Syerston lateritic Ni-Co mineralisation. The proposed process involves high pressure acid leaching followed by decantation of pregnant liquor by counter current decantation, neutralisation and mixed sulphide precipitation.

SRK are not aware of any metallurgical factors that might preclude the deposit from being reported as an identified Mineral Resource.

Metallurgical assumptions involved in converting nickel and cobalt grades to a combined nickel equivalent value were specified by Scandium21 on the basis of the previous feasibility studies. They were discussed in Section 13. They remained unchanged from those used in 2005.

## Environmental assumptions

As far as McDonald Speijers is aware, the area in which the nickel- cobalt resource occurs has no unusual environmental significance.

Feasibility studies have developed plans for topsoil removal and stockpiling and for waste dumps. Mine waste should not contain any sulphide minerals and should be chemically inert.

Process plant tailings would be neutralised and disposed of in conventional dams and evaporation ponds suitable for local climatic conditions. Feasibility study work included design of a suitable tailings disposal facility.

An EIS was prepared in parallel with the 2000 FS and in May 2001 the proposed Ni-Co project received Development Consent under the NSW *Environmental Planning and Assessment Act*.

The Development Consent defined environmental requirements and set conditions that included:

- Environmental management plans
- Monitoring requirements
- Environmental reporting

- Specific environmental mitigation measures
- Emission limits
- Studies and plans for hazard and risk control
- Appointment of an Environmental Officer
- Rehabilitation.

In 2005, Ivanplats submitted an application to modify the Development Consent to allow for some changes including an increase in proposed plant throughput. This was approved in October 2005.

Although a Ni-Co project did not proceed at the time, McDonald Speijers understands that Scandium21 was advised that sufficient preparatory work was done to trigger the consent.

Despite the fact that additional permits and licences would have to be obtained before operations could commence, granting of a Development Consent indicates that there are unlikely to be any insurmountable environmental obstacles.

There are no obvious environmental factors that would prevent the deposit being reported as an Identified Mineral Resource.

### 14.1.4 2016 Mineral Resource estimate

The resource estimate was obtained by evaluating the new block model at the estimated operating mill feed breakeven cut-off grade for all zones below the base of the transported alluvium.

At a cut-off grade of 0.6% Nickel equivalent, using data available to us as at 1 February 2016, total Ni-Co resources are estimated to be approximately as shown in Table 14-8.

NiEq% = Ni% + (Co% * 2.95) assuming nickel at USD4/lb, cobalt at USD12/lb with nickel recovery of 90% and cobalt recovery of 88.9%

**Table 14-8: Summary resource estimate - 0.60% NiEq cut-off**

| Cut-off NiEq% | Class | Inventory (Mt) | NiEq (%) | Cont. Metal (NiEq kt) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|---|---|
| 0.6 | Measured | 52 | 1.05 | 540 | 0.73 | 380 | 0.11 | 57 |
| 0.6 | Indicated | 49 | 0.87 | 430 | 0.58 | 280 | 0.10 | 49 |
| 0.6 | Meas + Ind | 101 | 0.97 | 970 | 0.65 | 660 | 0.10 | 106 |
| 0.6 | Inferred | 8 | 0.83 | 700 | 0.54 | 50 | 0.10 | 8 |

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes and contained metal are rounded to the nearest thousand.
3) Totals may appear different from the sum of their components due to rounding.
4) A cut-off grade of 0.6% NiEq, NiEq% = Ni% + (Co% x 2.95)
5) USD4/lb, cobalt at USD12/lb and a nickel recovery of 90% and cobalt recovery of 88.9%, USD:AUD of 0.75.
6) The Mineral Resource estimation was verified by Danny Kentwell, FAusIMM, who is a full-time employee of SRK Consulting. Danny Kentwell, FAusIMM, a full-time employee of SRK Consulting is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

An evaluation of the resource block model broken down by zone and classification is summarised in Table 14-9. A summary of global block model results at a range of NiEq cut-offs is shown in Table 14-10.

**Table 14-9: Summary of block model tonnages & grades by zone, 0.60% NiEq cut-off, all elements**

| Cut-off NiEq (%) | Zone | Class | Inventory (Mt) | NiEq (%) | Grade (% Ni) | Grade (% Co) |
|---|---|---|---|---|---|---|
| 0.6 | OVB | Measured | 0.04 | 0.83 | 0.24 | 0.198 |
| 0.6 | OVB | Indicated | 0.04 | 0.67 | 0.40 | 0.092 |
| 0.6 | OVB | *Meas + Ind* | 0.08 | 0.74 | 0.32 | 0.142 |
| 0.6 | OVB | Inferred | 0.00 | | | |
| 0.6 | TZ | Measured | 6.6 | 0.80 | 0.62 | 0.061 |
| 0.6 | TZ | Indicated | 6.9 | 0.76 | 0.56 | 0.066 |
| 0.6 | TZ | *Meas + Ind* | 13.5 | 0.78 | 0.59 | 0.064 |
| 0.6 | TZ | Inferred | 0.8 | 0.74 | 0.50 | 0.080 |
| 0.6 | GZ | Measured | 21.7 | 1.25 | 0.79 | 0.156 |
| 0.6 | GZ | Indicated | 19.6 | 0.98 | 0.58 | 0.136 |
| 0.6 | GZ | *Meas + Ind* | 41.3 | 1.12 | 0.69 | 0.147 |
| 0.6 | GZ | Inferred | 4.7 | 0.89 | 0.56 | 0.113 |
| 0.6 | SGZ | Measured | 22.4 | 0.94 | 0.70 | 0.082 |
| 0.6 | SGZ | Indicated | 21.7 | 0.82 | 0.58 | 0.082 |
| 0.6 | SGZ | *Meas + Ind* | 44.2 | 0.88 | 0.64 | 0.082 |
| 0.6 | SGZ | Inferred | 2.5 | 0.78 | 0.53 | 0.084 |
| 0.6 | SAP | Measured | 1.0 | 0.76 | 0.67 | 0.031 |
| 0.6 | SAP | Indicated | 0.5 | 0.73 | 0.60 | 0.044 |
| 0.6 | SAP | *Meas + Ind* | 1.5 | 0.75 | 0.64 | 0.036 |
| 0.6 | SAP | Inferred | 0.4 | 0.69 | 0.57 | 0.040 |
| 0.6 | TOTAL | Measured | 51.7 | 1.05 | 0.73 | 0.109 |
| 0.6 | TOTAL | Indicated | 48.8 | 0.87 | 0.58 | 0.101 |
| 0.6 | TOTAL | *Meas + Ind* | 100.5 | 0.97 | 0.65 | 0.105 |
| 0.6 | TOTAL | Inferred | 8.4 | 0.83 | 0.54 | 0.098 |

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources Modified to produce Mineral Reserves.

NiEq% = Ni% + (Co% * 2.95) assuming nickel at USD4/lb, cobalt at USD12/lb with nickel recovery of 90% and cobalt recovery of 88.9%

**Table 14-10: Summary of global block model evaluations at selected cut-off grades**

| Cut-off NiEq (%) | Zone | Class | Inventory (Mt) | NiEq (%) | Grade (% Ni) |
|---|---|---|---|---|---|
| 0.6 | Measured | 51.5 | 1.06 | 0.73 | 0.111 |
| 0.6 | Indicated | 48.8 | 0.88 | 0.58 | 0.101 |
| 0.6 | *Meas+Ind* | 100.3 | 0.97 | 0.65 | 0.106 |
| 0.6 | Inferred | 8.5 | 0.83 | 0.54 | 0.098 |
| 0.6 | *Total* | 108.7 | 0.96 | 0.65 | 0.105 |
| 0.8 | Measured | 37.1 | 1.20 | 0.81 | 0.131 |
| 0.8 | Indicated | 26.5 | 1.03 | 0.66 | 0.126 |

| Cut-off NiEq (%) | Zone | Class | Inventory (Mt) | NiEq (%) | Grade (% Ni) |
|---|---|---|---|---|---|
| 0.8 | *Meas+Ind* | 63.6 | 1.13 | 0.75 | 0.13 |
| 0.8 | Inferred | 3.8 | 1.01 | 0.66 | 0.118 |
| 0.8 | *Total* | 67.4 | 1.12 | 0.74 | 0.13 |
| 1.0 | Measured | 24.7 | 1.35 | 0.90 | 0.154 |
| 1.0 | Indicated | 11.7 | 1.21 | 0.74 | 0.158 |
| 1.0 | *Meas+Ind* | 36.3 | 1.31 | 0.85 | 0.16 |
| 1.0 | Inferred | 1.7 | 1.16 | 0.75 | 0.137 |
| 1.0 | *Total* | 38.1 | 1.30 | 0.84 | 0.15 |

Note: All Zones Below Base of Alluvium, By Resource Category.

NiEq% = Ni% + (Co% * 2.95) assuming nickel at USD4/lb, cobalt at USD12/lb with nickel recovery of 90% and cobalt recovery of 88.9%

### 14.1.5 Comparison with previous estimates

A comparison between the current estimate and block model estimates made in 2000 and 2005 is shown in Table 14-11.

At a cut-off grade of 0.60 NiEq% the additional drill hole data available in 2016 resulted in only a small increase in global tonnage at virtually identical average nickel and cobalt grades

A cut-off grade of 0.60 NiEq% roughly approximates 0.65 NiEq2000% in the 2000 resource model developed by SLA.

**Table 14-11: Comparison of block models, below base of alluvium**

| Cut-off NiEq (%) | Zone | Class | Inventory (Mt) | NiEq (%) | Cont. Metal (NiEq kt) | Grade (% Ni) | Cont. Metal (Ni kt) | Grade (% Co) | Cont. Metal (Co kt) |
|---|---|---|---|---|---|---|---|---|---|
| Current Estimate | | | | | | | | | |
| 0.6 | Total | Measured | 52 | 1.05 | 540 | 0.73 | 380 | 0.109 | 57 |
| 0.6 | Total | Indicated | 49 | 0.87 | 430 | 0.58 | 280 | 0.101 | 49 |
| 0.6 | Total | Meas+Ind | 101 | 0.97 | 970 | 0.65 | 660 | 0.105 | 106 |
| 0.6 | Total | Inferred | 8 | 0.83 | 70 | 0.54 | 50 | 0.098 | 8 |
| 0.6 | Total | Total | 109 | 0.95 | 1,040 | 0.65 | 700 | 0.105 | 114 |
| Previous Estimate | | | | | | | | | |
| 0.6 | Total | Measured | 52 | 1.06 | 550,000 | 0.73 | 380 | 0.111 | 58 |
| 0.6 | Total | Indicated | 47 | 0.88 | 420,000 | 0.59 | 280 | 0.1 | 47 |
| 0.6 | Total | Meas+Ind | 99 | 0.98 | 970,000 | 0.66 | 660 | 0.106 | 105 |
| 0.6 | Total | Inferred | 8 | 0.84 | 70,000 | 0.54 | 40 | 0.098 | 8 |
| 0.6 | Total | Total | 107 | 0.97 | 1,030 | 0.65 | 700 | 0.105 | 113 |
| Variance | | | | | | | | | |
| 0.6 | Total | Measured | *-0.2%* | *-0.9%* | *-1.1%* | *0.0%* | *-1.0%* | *-1.8%* | *-1.6%* |
| 0.6 | Total | Indicated | *3.6%* | *-1.1%* | *2.5%* | *-1.7%* | *1.6%* | *1.0%* | *4.4%* |
| 0.6 | Total | Meas+Ind | *1.6%* | *-1.0%* | *0.5%* | *-1.5%* | *0.1%* | *-0.9%* | *1.1%* |
| 0.6 | Total | Inferred | *2.4%* | *-1.2%* | *1.5%* | *0.0%* | *1.3%* | *0.0%* | *1.1%* |
| 0.6 | Total | Total | *1.7%* | *-2.1%* | *0.5%* | *0.0%* | *0.2%* | *0.0%* | *1.1%* |

NiEq% = Ni% + (Co% * 2.95) assuming nickel at USD4/lb, cobalt at USD12/lb with nickel recovery of 90% and cobalt recovery of 88.9%. At the same, NiEq 2000 cut-off grade the 2005 global estimate was within about ±10% of the 2000 estimate by SLA, indicating that global resource estimates were reasonably robust. The largest difference was in cobalt grade, which dropped by about 11%, mainly due to:

- No longer interpreting the top of the GZ on the basis of high cobalt grades
- The 2005 program of infill RC drilling in areas of unusually high cobalt grades, which showed them to have poorer continuity than previously presumed
- The application of harsher cobalt top-cuts in areas still delineated by drilling at spacings of 60 m or more.

However, the lower average cobalt grade was partly offset by an increase in estimated tonnage of about 8%.

### 14.1.6 Resource risk

Mineral resource estimates are not precise calculations, being dependent on the interpretation of limited information about the location, shape and continuity of the mineral occurrence and on the reliability of available sampling results.

The principal factors that may contribute to resource estimation error are:

Limited lateral continuity of unusually high cobalt grades. While there has been considerable lateral dispersion of low to moderate cobalt values, infill and twin drilling carried out in 2005 showed that high cobalt values (above about 0.4% to 0.5%) can have very limited lateral continuity. They may be associated with Mn concentrations in relict structures that are not sub-horizontal and may therefore be poorly represented in vertical drill holes. This can result in serious overestimation of cobalt grades around unusually high grade intercepts in widely spaced, pre-development drill holes. Local infill drilling to 30 x 30 m in 2005 and harsher cobalt top-cuts in areas of wider-spaced drilling have ameliorated this risk, but it has not been eliminated.

Potential error in bulk density factors. These are based on limited numbers of direct measurements, which might not prove to be fully representative of the deposit as a whole.

Any significant increase in cut-off grade would increase risk levels due to block estimate smoothing inherent in the ordinary kriging estimation process.

## 14.2 2017 Mineral Resource estimate

### 14.2.1 Accepted data

Micromine format collar, survey, assay and geological coding data was received by Widenbar and Associates on 29 August 2017.

The database consisted of a total 1,502 drill holes for a total of 55,611 metres. 1,354 holes were Reverse Circulation (RC), with the remainder (148) being Air Core (AC) holes. No RAB holes were included in the database because of potential contamination and lack of confidence in collar coordinates. AC holes were subject to doubt because:

- Collar coordinates could not be properly validated from first principles from the original surveyor's records
- There was a slight bias between AC and RC holes due to the ALS original results with the UT check re-assay program
- Many failed to fully penetrate the laterite profile.

AC holes were only accepted in areas where there was no effective coverage by the RC drilling pattern. If an adjacent RC hole was present, this was used in preference if a AC hole is present.

Calweld holes were also excluded because of their extreme difference in sample support and because they tended not to penetrate the complete laterite profile.

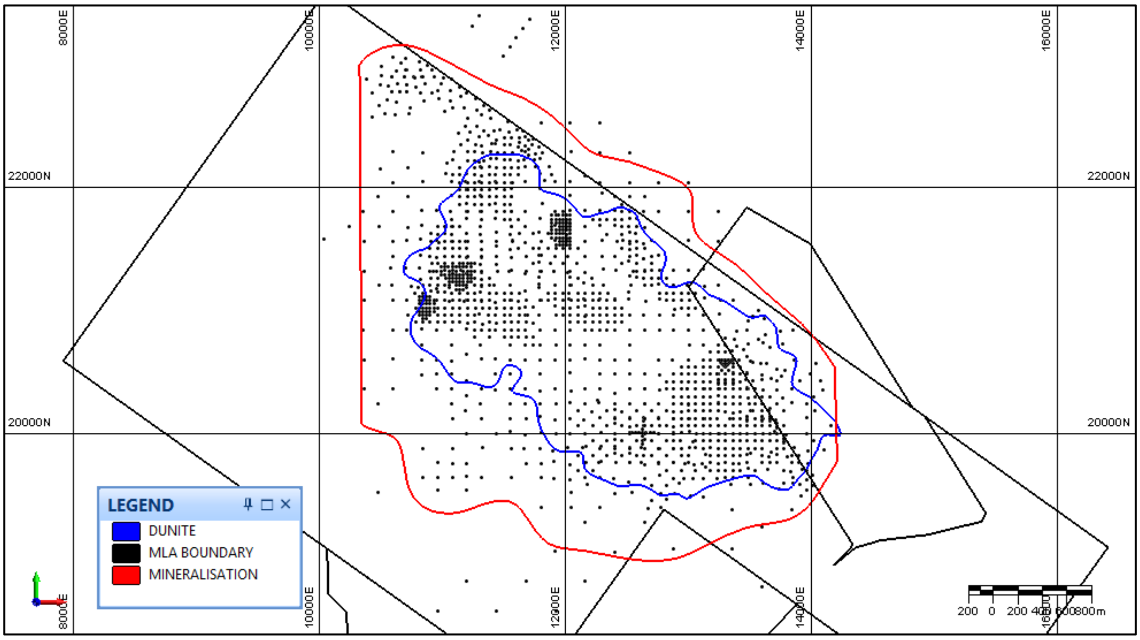



**Figure 14-9: Drill hole location plan**

### 14.2.2 Sample lengths and compositing

A histogram of raw assay lengths is shown below. On the basis that the majority of samples are one metre in length, it was decided to composite the assay data to 1m. All composites had a length of 1m.

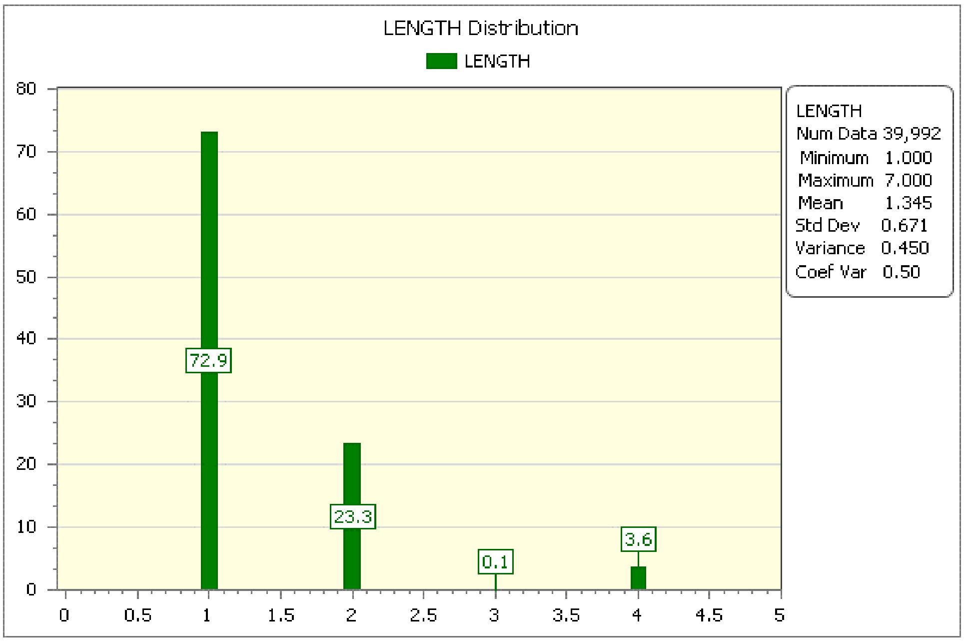



**Figure 14-10: Assay sample length histogram**

### 14.2.3 Statistical analysis

All statistical analysis is carried out on 1m composite data, coded by geological domain (LATZONE). Summary statistics of the main elements of economic interest are tabulated below.

**Table 14-12: Summary statistics of major elements**

| Extended Statistics for Composites_1 | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Ni_pct | Co_pct | Sc_ppm | Pt_ppm | Fe_pct | Al_pct | Si_pct | Mg_pct |
| NORMAL STATS | | | | | | | | |
| Mean | 0.404 | 0.064 | 70.264 | 0.162 | 28.465 | 3.474 | 17.067 | 2.238 |
| Median | 0.304 | 0.026 | 39 | 0.07 | 29.8 | 2.8 | 17 | 0.65 |
| Std Dev | 0.372 | 0.102 | 100.57 | 0.535 | 14.827 | 2.979 | 10.698 | 4.055 |
| Variance | 0.139 | 0.01 | 10,114.30 | 0.286 | 219.825 | 8.873 | 114.441 | 16.446 |
| Std Error | 0.002 | 0 | 0.484 | 0.002 | 0.071 | 0.014 | 0.06 | 0.019 |
| Coeff Var | 0.92 | 1.589 | 1.431 | 3.294 | 0.521 | 0.858 | 0.627 | 1.812 |
| Minimum | 0 | 0 | 1 | 0 | 0.2 | 0.001 | 0.001 | 0.02 |
| Maximum | 4.19 | 2.72 | 900 | 30 | 60.1 | 16.8 | 42 | 29.2 |
| | | | | | | | | |
| DATA | | | | | | | | |
| Total data | 53,796 | 53,796 | 53,796 | 53,796 | 53,796 | 53,796 | 53,796 | 53,796 |
| Valid data | 50,164 | 50,164 | 43,143 | 47,759 | 43,278 | 42,946 | 31,963 | 43,456 |
| Missing data | 3,632 | 3,632 | 10,653 | 6,037 | 10,518 | 10,850 | 21,833 | 10,340 |
| Zero values | 6 | 6 | 0 | 0 | 0 | 0 | 0 | 0 |
| Negative values | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | | | | | | | | |
| PERCENTILES | | | | | | | | |
| 2.5 | 0.007 | 0.001 | 4 | 0.001 | 4.6 | 0.001 | 2 | 0.14 |
| 5 | 0.013 | 0.002 | 6 | 0.001 | 5.6 | 0.06 | 3 | 0.18 |
| 10 | 0.024 | 0.004 | 10 | 0.009 | 7.4 | 0.2 | 4 | 0.26 |
| 20 | 0.062 | 0.007 | 18.5 | 0.02 | 12 | 0.6 | 6 | 0.36 |
| 30 | 0.121 | 0.011 | 26 | 0.03 | 17.5 | 1.2 | 8 | 0.46 |
| 40 | 0.194 | 0.017 | 33 | 0.049 | 23.9 | 1.9 | 12 | 0.55 |
| 50 | 0.304 | 0.026 | 39 | 0.07 | 29.8 | 2.8 | 17 | 0.65 |
| 60 | 0.434 | 0.04 | 49 | 0.1 | 34.9 | 3.8 | 21 | 0.8 |
| 70 | 0.564 | 0.062 | 63 | 0.15 | 39.6 | 4.9 | 24 | 1.19 |
| 80 | 0.712 | 0.093 | 86 | 0.224 | 43.8 | 6.11 | 27 | 2.33 |
| 90 | 0.926 | 0.169 | 140 | 0.378 | 47.6 | 7.8 | 32 | 6.75 |
| 95 | 1.12 | 0.258 | 270.85 | 0.535 | 49.8 | 9.1 | 35 | 12.5 |
| 97.5 | 1.29 | 0.349 | 408.425 | 0.739 | 51.3 | 10.2 | 38 | 16.8 |
| 98 | 1.34 | 0.378 | 439 | 0.81 | 51.7 | 10.6 | 38 | 17.7 |
| 99 | 1.5 | 0.494 | 556 | 1.06 | 52.8 | 11.4 | 39 | 19.6 |

## 14.2.4 Assay distribution analysis

Analysis of the distribution of the main elements has been carried out on a domain (LATZONE) basis. The aim is to ensure that domains are valid and differentiated correctly and to ensure that hard or soft boundaries can be applied appropriately.

Overlays of log probability plots by LATZONE are illustrated in the Figures below, with commentary following.

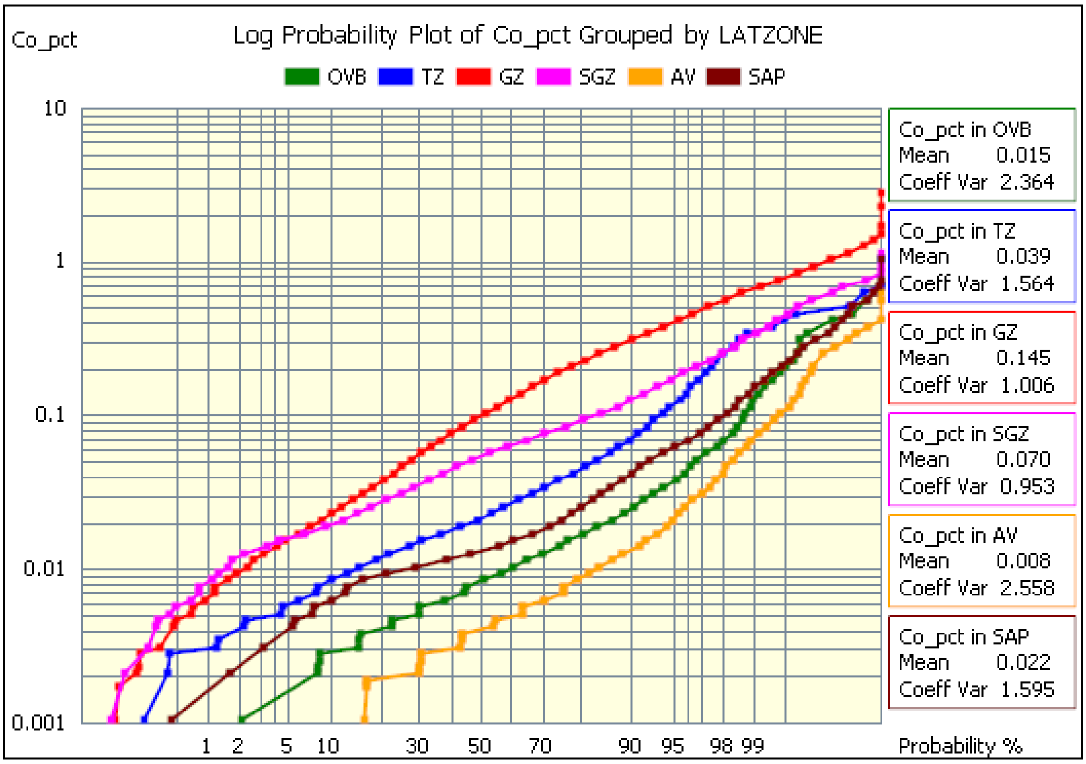



**Figure 14-11: Cobalt distribution**

The distribution of Cobalt is clearly very different in each of the domains.

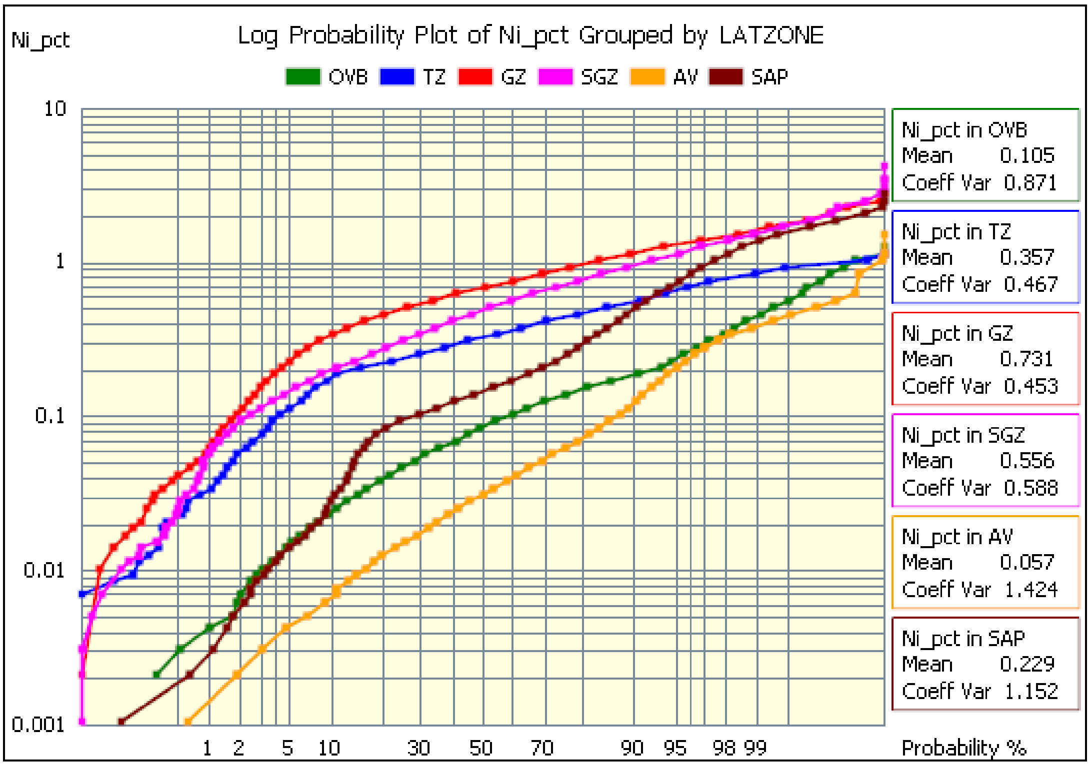



**Figure 14-12: Ni distribution**

As for Cobalt, the distribution of Ni is clearly very different in each of the domains.

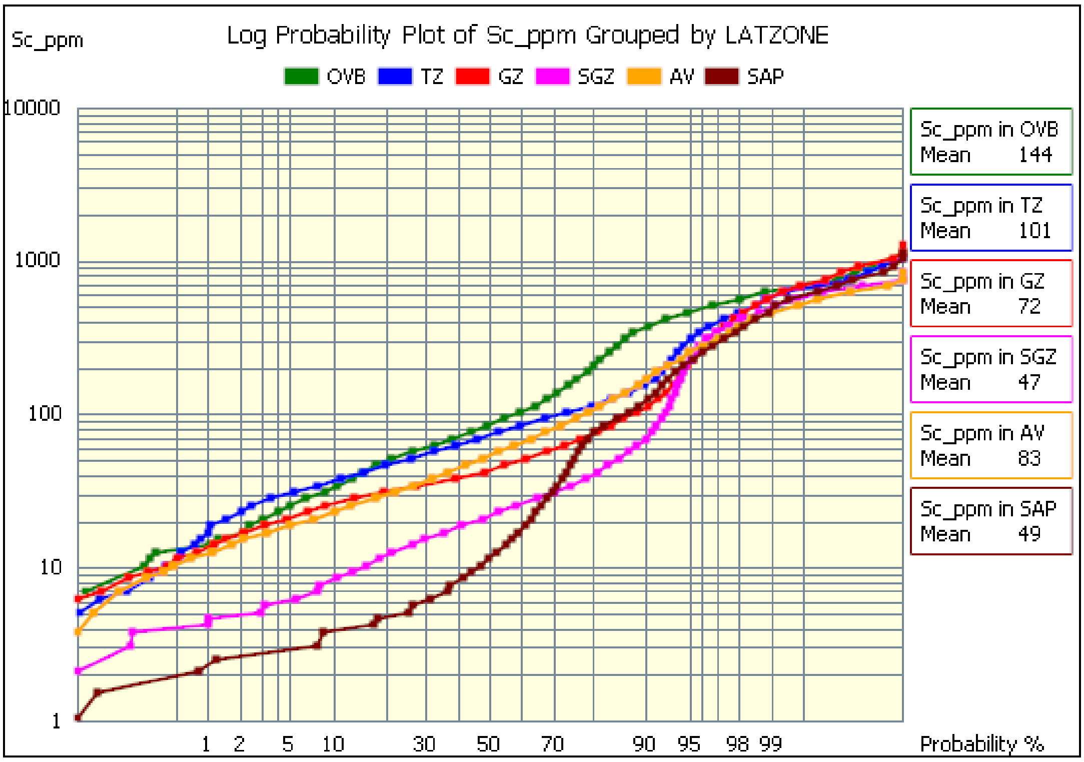



**Figure 14-13: Scandium distribution**

Similarly, Scandium shows clearly different distributions in each of the domains. However, there are also multiple distributions evident, as illustrated in the histograms below.

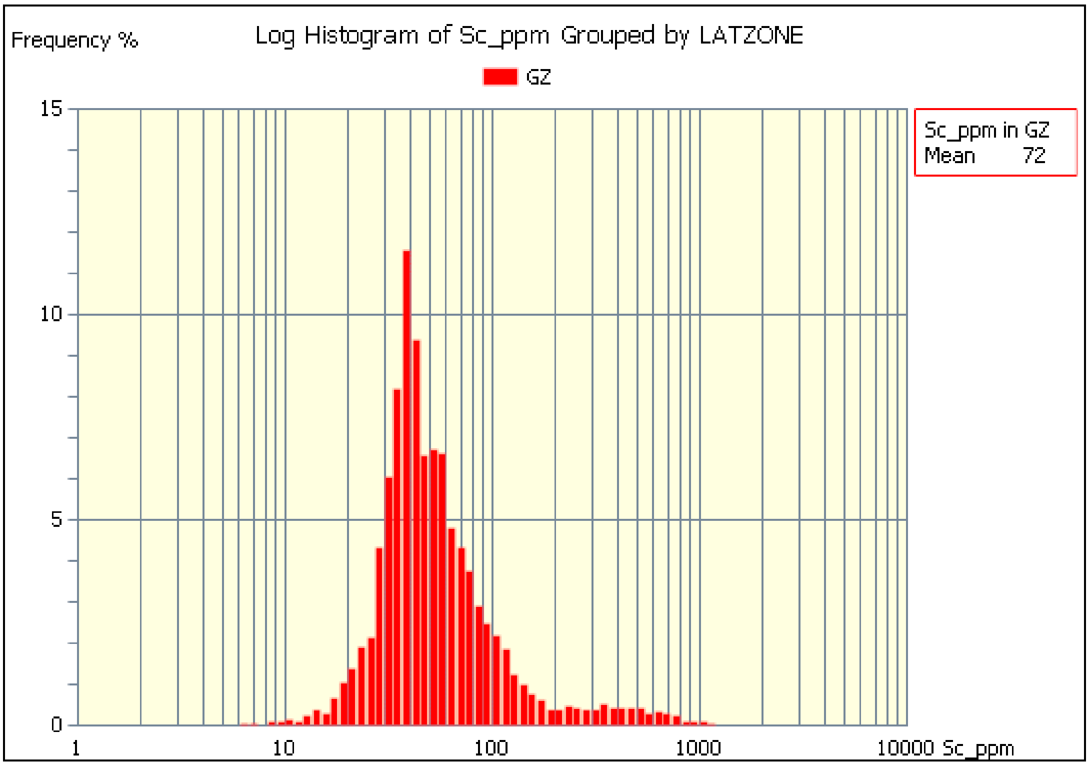



**Figure 14-14: Scandium in GZ**

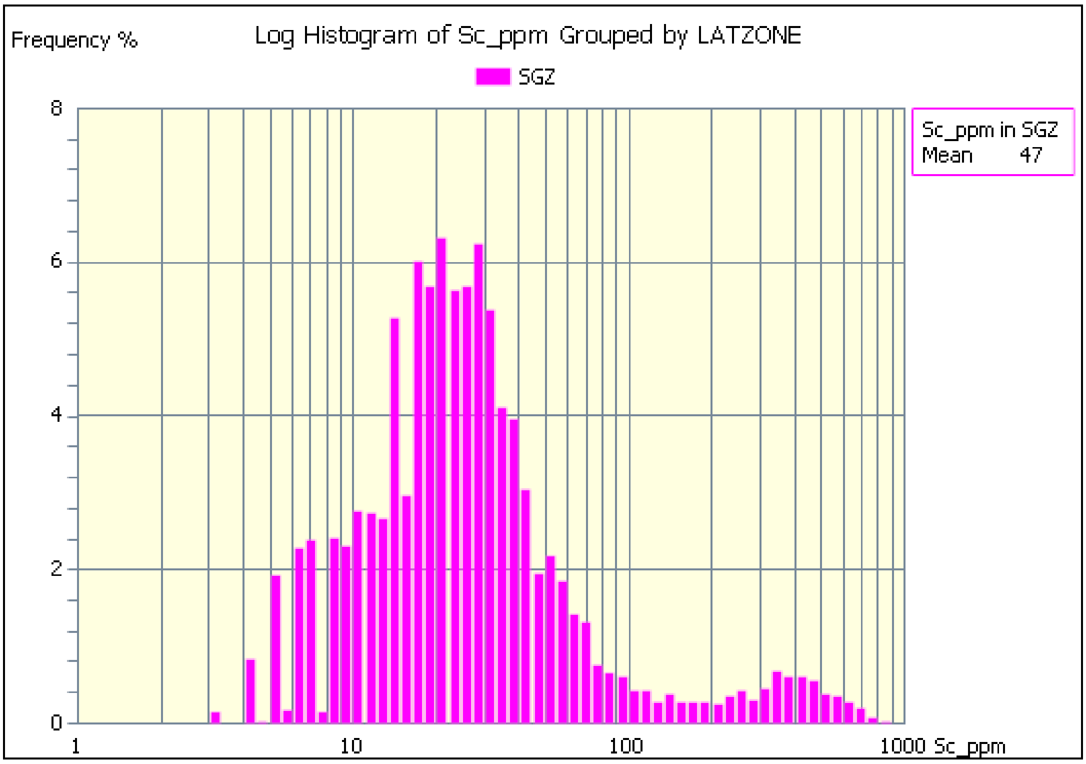



**Figure 14-15: Scandium in SGZ**

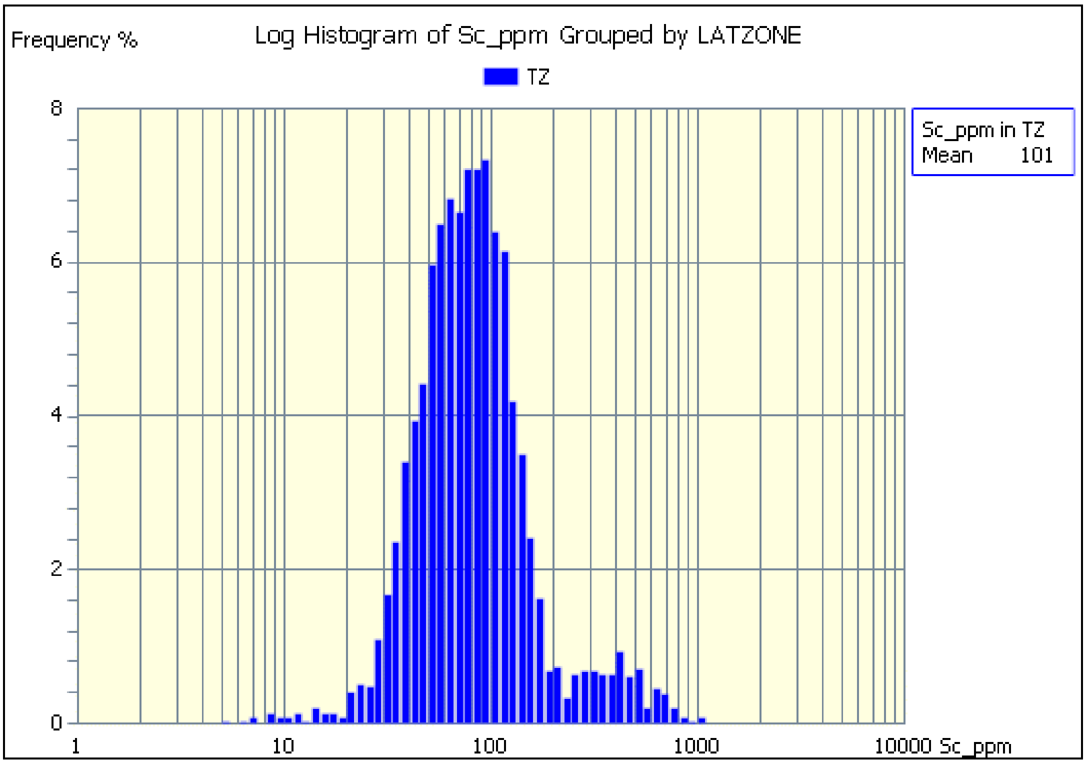



**Figure 14-16: Scandium in TZ**

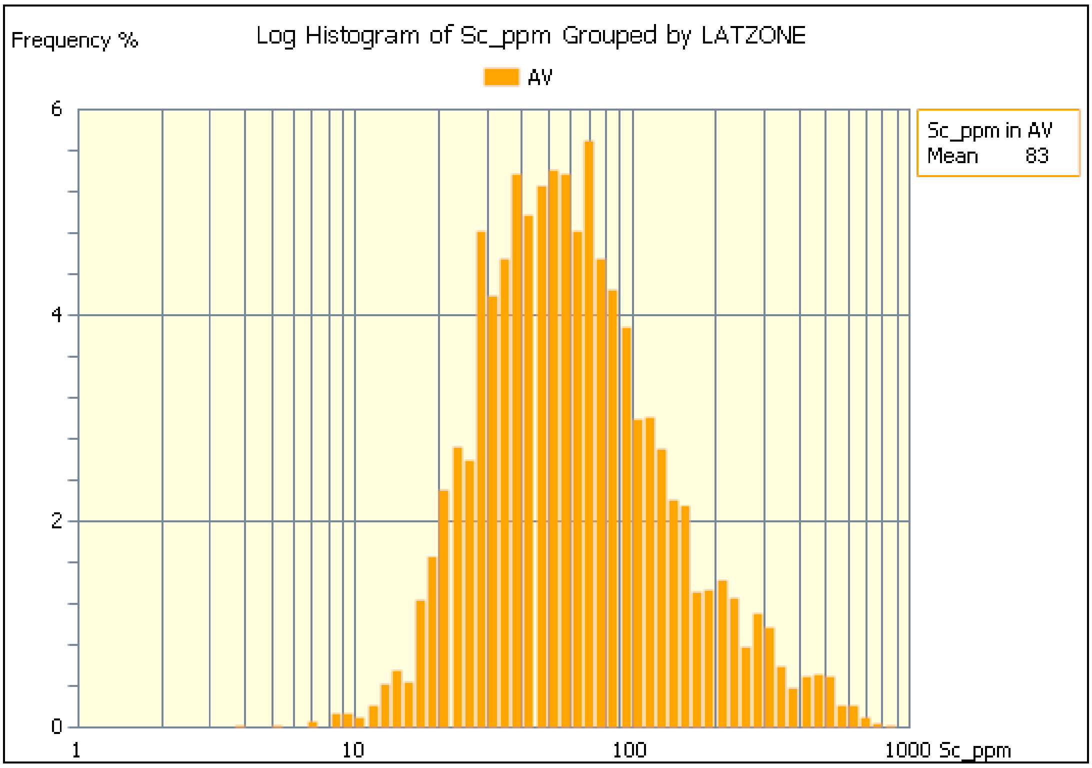



**Figure 14-17: Scandium in AV**

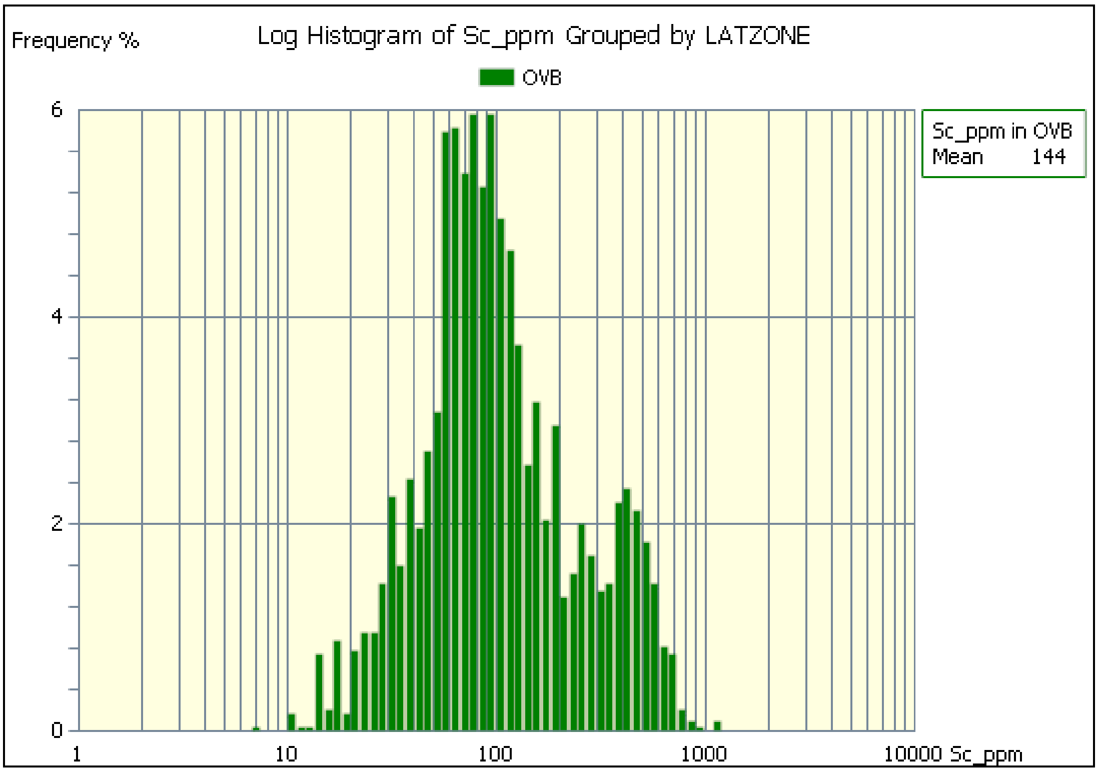



**Figure 14-18: Scandium in OVB**

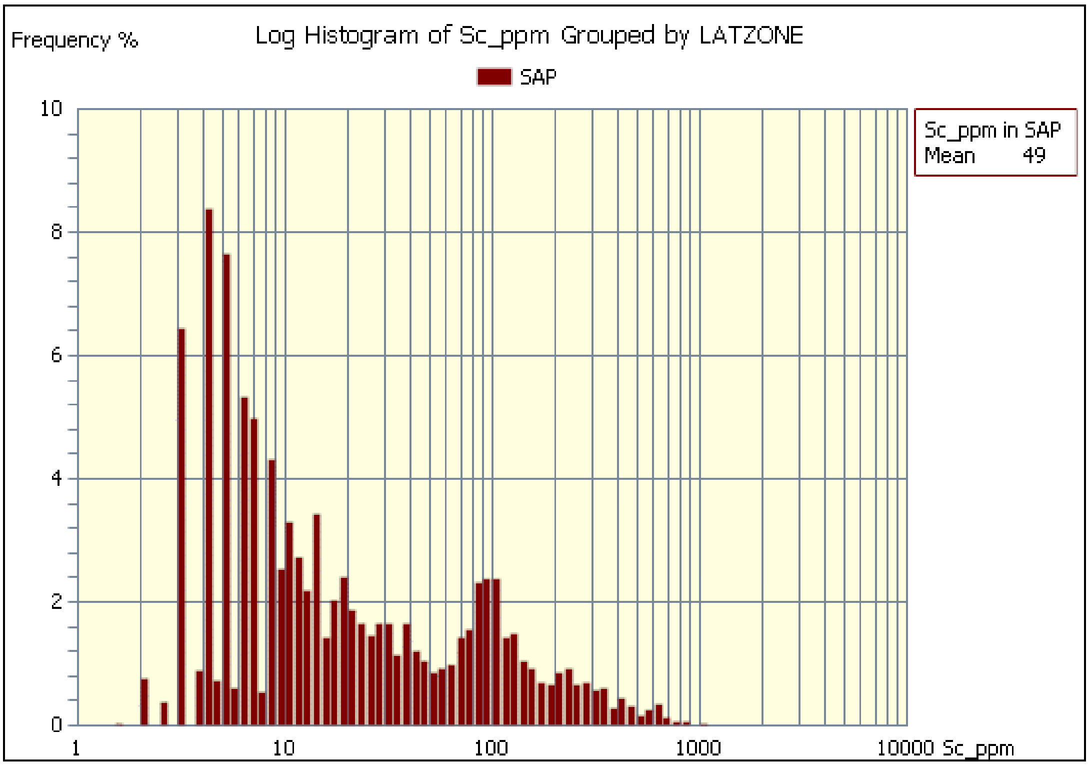



**Figure 14-19: Scandium in SAP**

As a result of the multiple population issue, an analysis was carried out on the spatial distribution of Scandium, both globally and by LATZONE domains. Scandium values were averaged over drill holes (for the global analysis) and over each LATZONE within a drill hole for the domain analysis. The results are illustrated below, and show clearly how the higher grade Scandium is concentrated outside the main Dunite complex outline. This is addressed in the resource model by defining high grade Scandium domains, which will be used to control and refine the modelling of Scandium. This is described in detail in Section 14.2.13 on Scandium modelling.

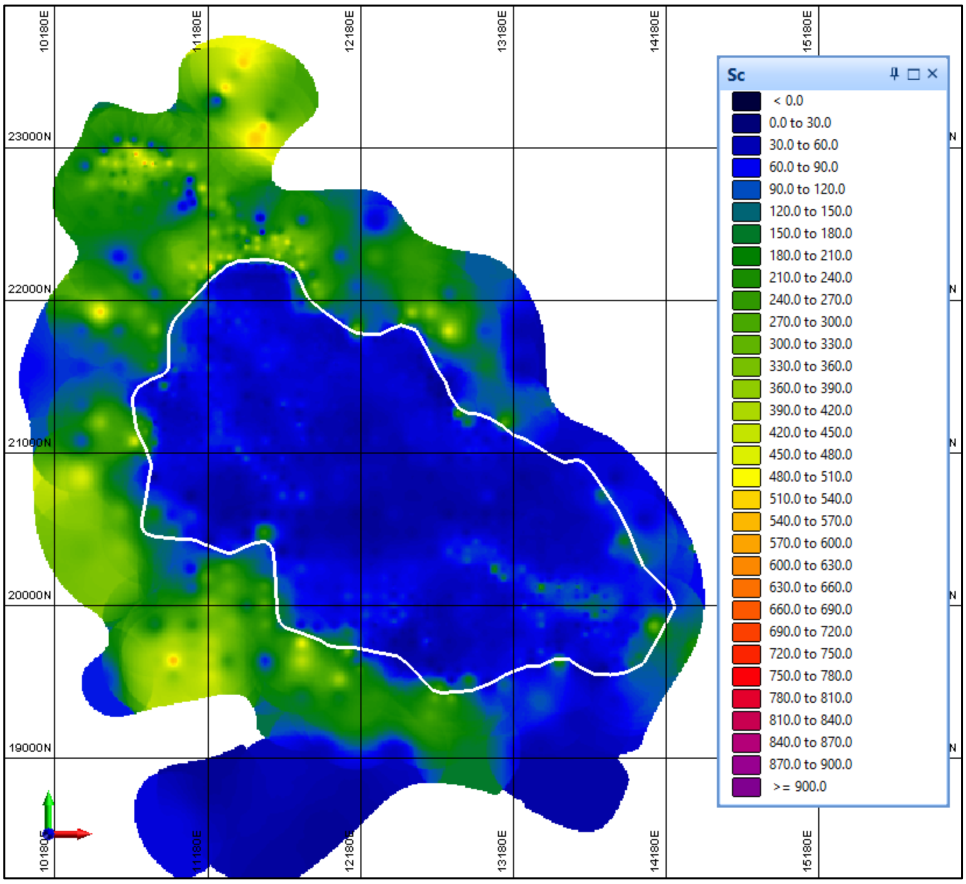



**Figure 14-20: Global scandium distribution**

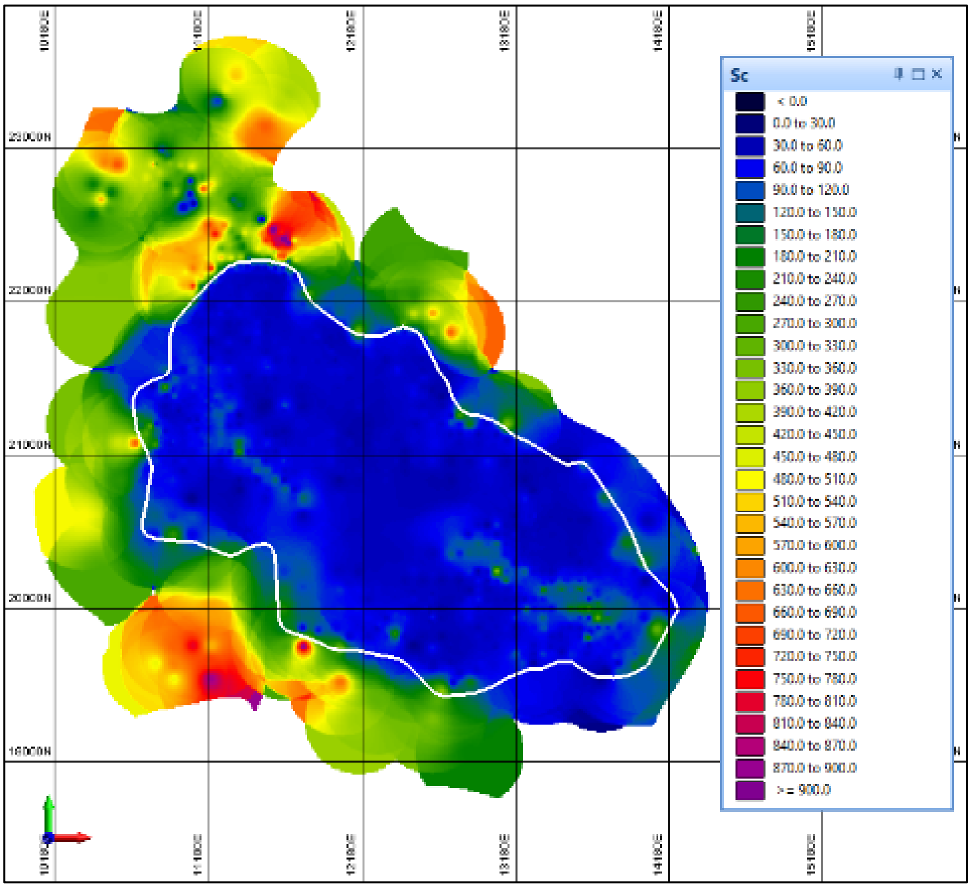



**Figure 14-21: Scandium in GZ**

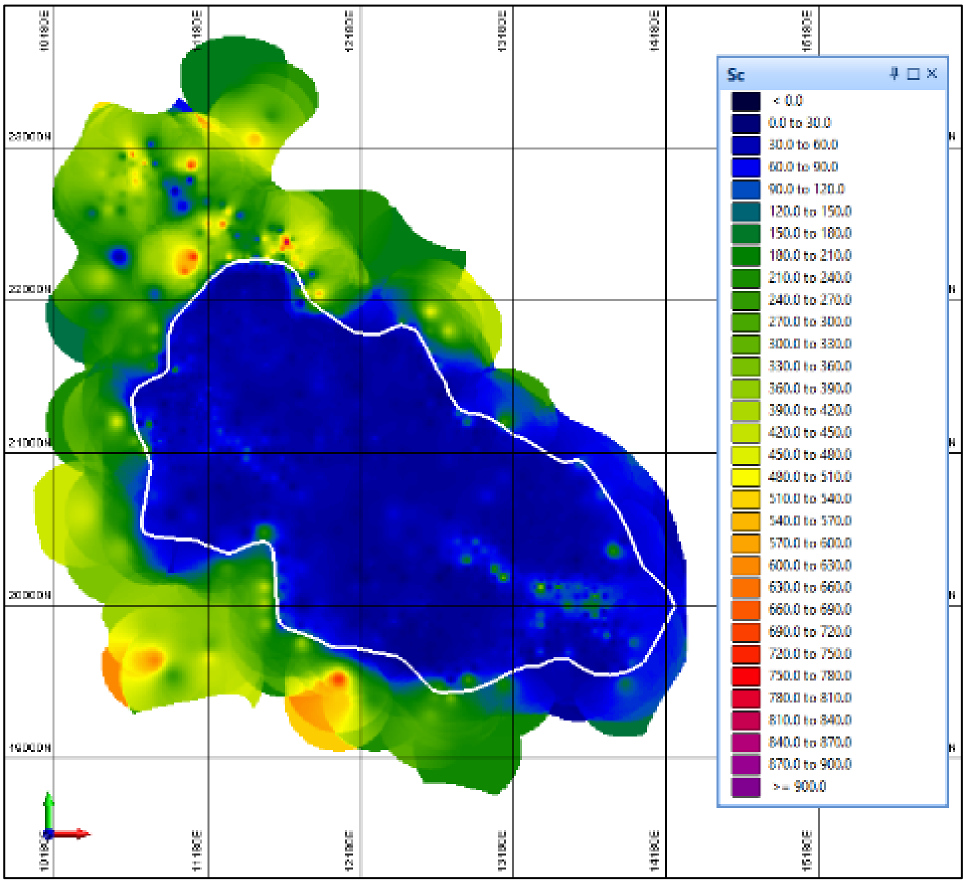



**Figure 14-22: Scandium in SGZ**

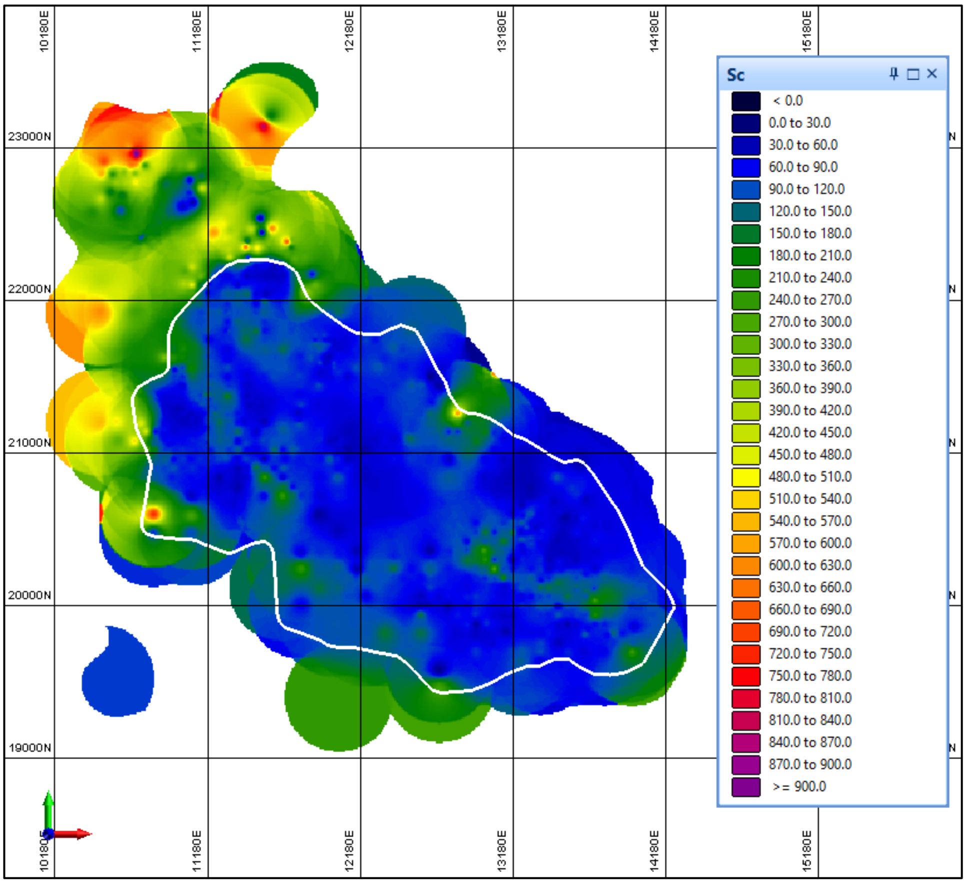



**Figure 14-23: Scandium in TZ**

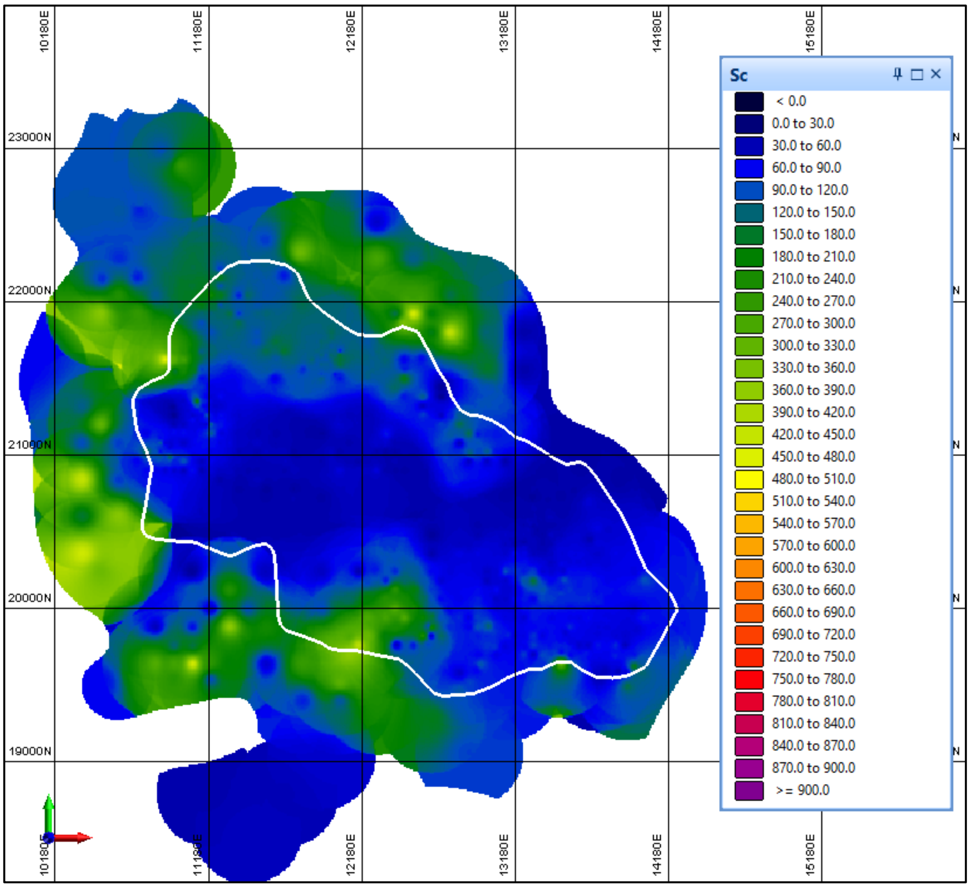



**Figure 14-24: Scandium in AV**

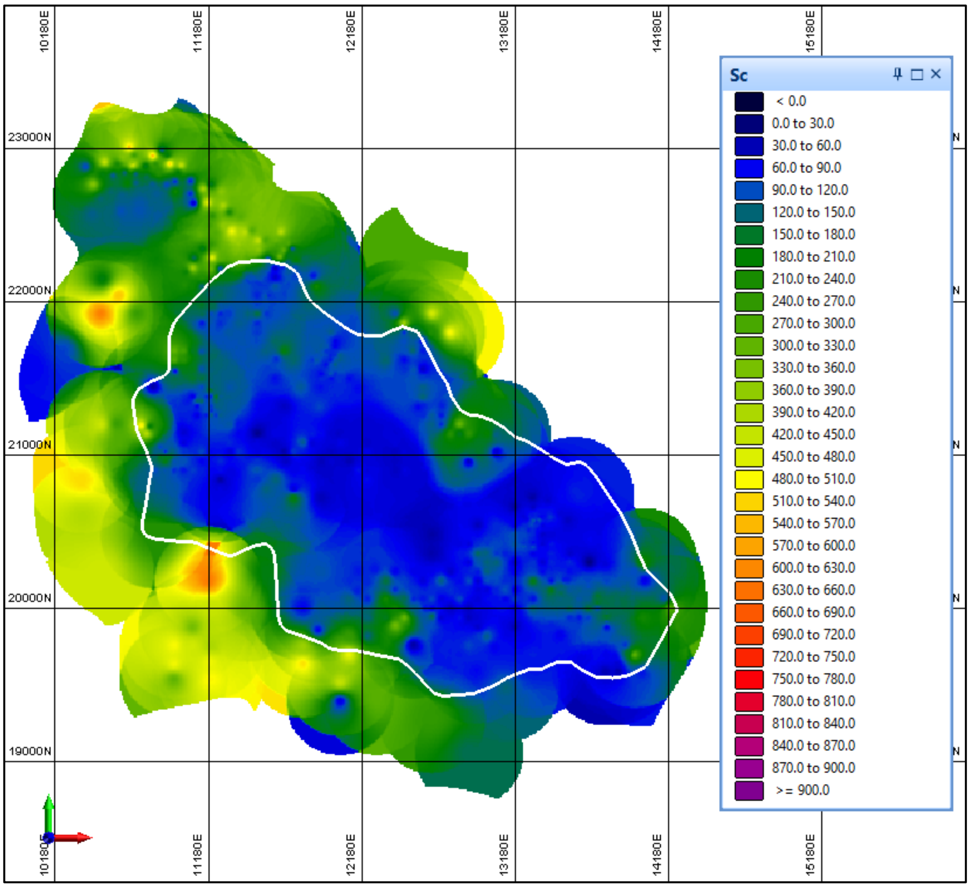



**Figure 14-25: Scandium in OVB**

When Scandium distribution is separated into data above the Dunite complex, and data outside the Dunite complex, the differences are clear.

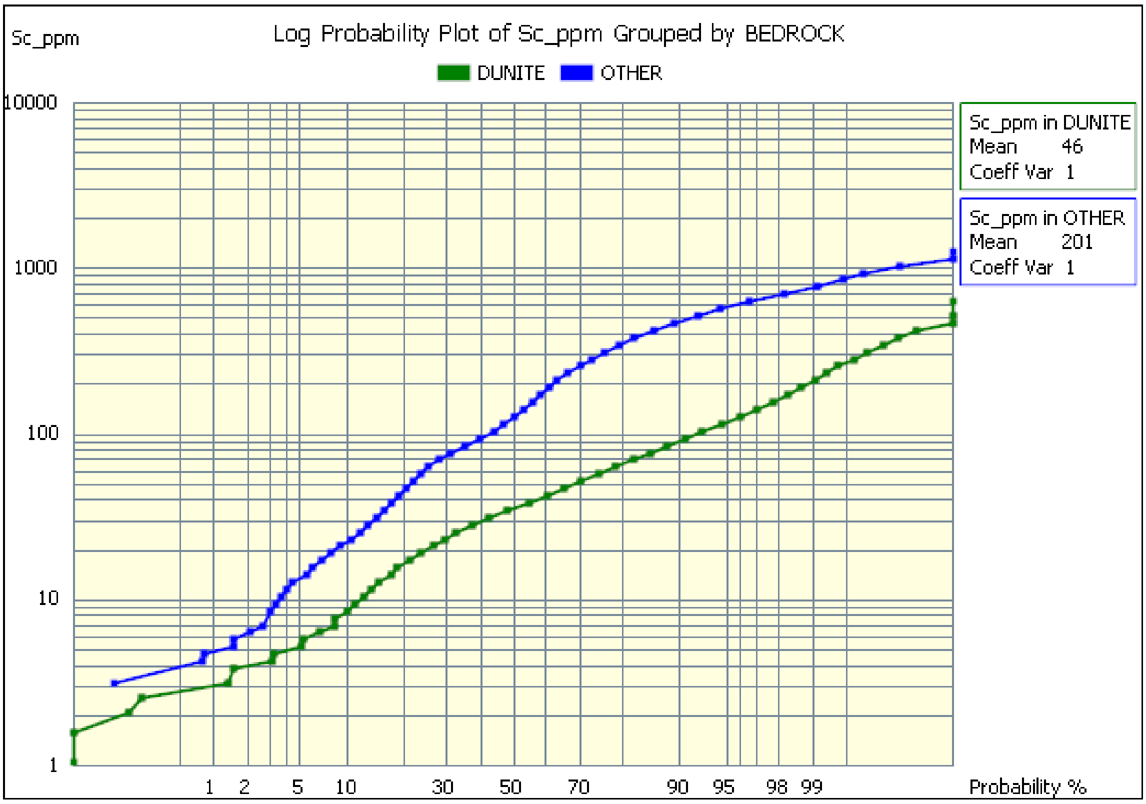



**Figure 14-26: Scandium above Dunite complex**

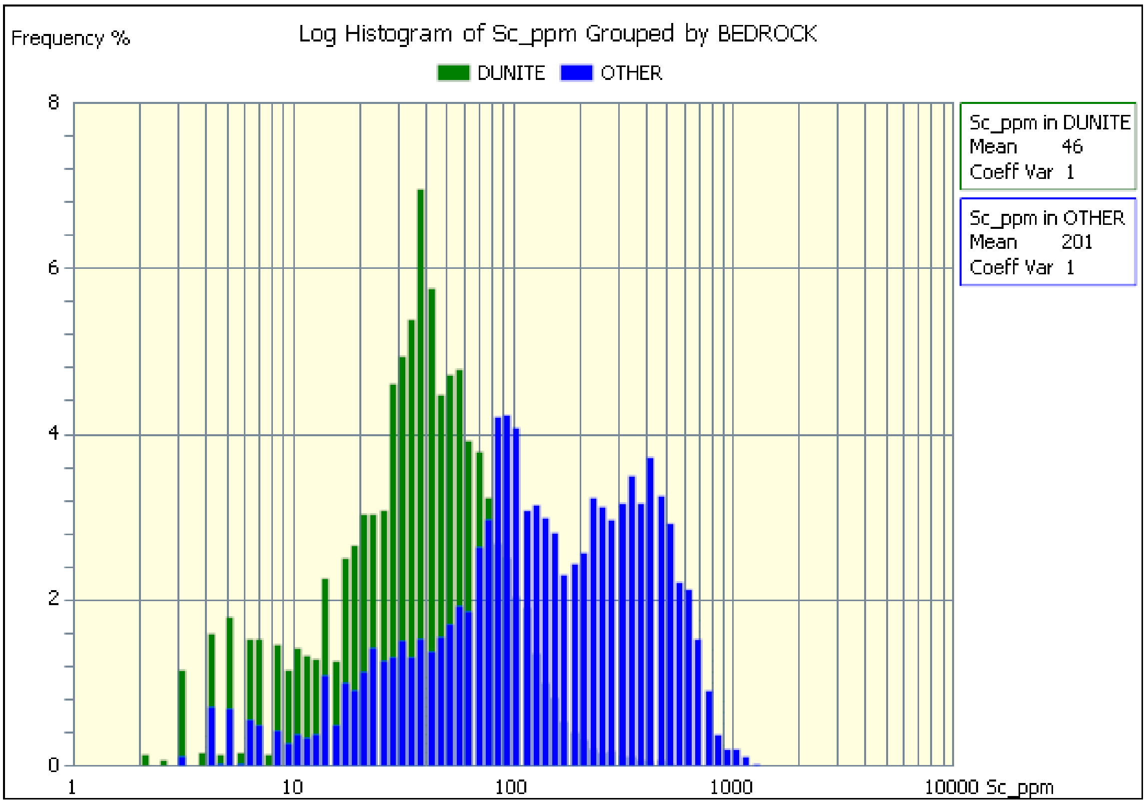



**Figure 14-27: Data outside Dunite complex**

It is also clear that there are multiple populations in the material outside the Dunite area, and this is addressed in the modelling by further high/low grade Scandium domaining in this area.

### 14.2.5 Assay caps

Caps have been applied on a domain (LATZONE) basis, for many of the variables prior to resource model estimation.

The intent of the cap is to reduce the effect of a very small number of extreme outliers. Unlike a gold deposit, which typically has a long tail of high grade values in the distribution which can materially affect grade estimation, there are very few outliers for most elements, and they are not extreme. The capping analysis has reviewed log probability plot and histogram distributions and applied appropriate caps where necessary. A summary of top cuts is shown in Table 14-13.

**Table 14-13: Applied top-cuts**

| HIGH GRADE CAPS BY LATZONE | | |
|---|---|---|
| LATZONE | Ni% | Co% |
| GZ | 2.50 | 1.30 |
| SGZ | 2.50 | 0.75 |
| TZ | 1.00 | 0.70 |
| AV | None | 0.40 |
| OVB | 1.25 | 0.40 |
| SAP | 2.00 | 0.70 |
| | | |
| Minor Variables | | |
| Sc_ppm | 900 | |
| Pt_ppm | 30 | |
| Pd_ppb | 600 | |
| Au_ppb | 600 | |
| Mn_ppm | 15000 | |
| Zn_ppm | 1500 | |
| Cu_ppm | 6000 | |
| Cr_ppm | 10000 | |
| As_ppm | 70 | |
| | | |
| Others Uncut | | |

The overlay log probability plots shown in Figure 14-28 to Figure 14-40 show the basis of the caps and also again illustrate that the LATZONE domaining for almost all elements shows clearly different distributions.

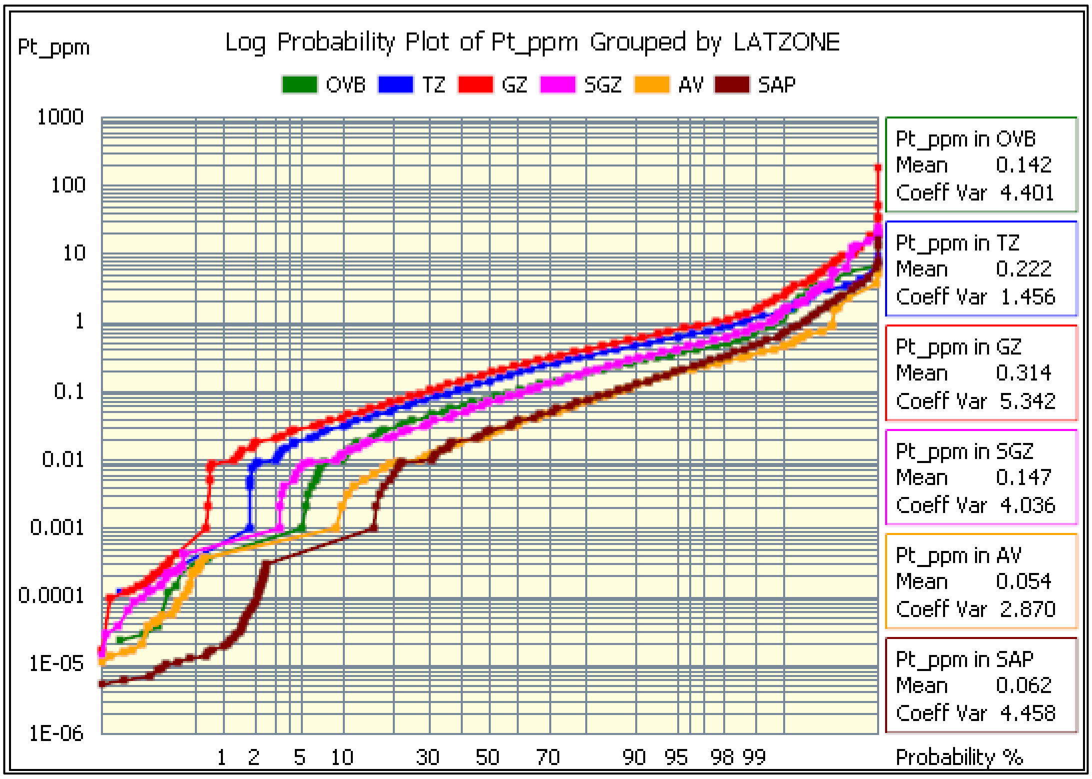



**Figure 14-28: Log probability plot of platinum**

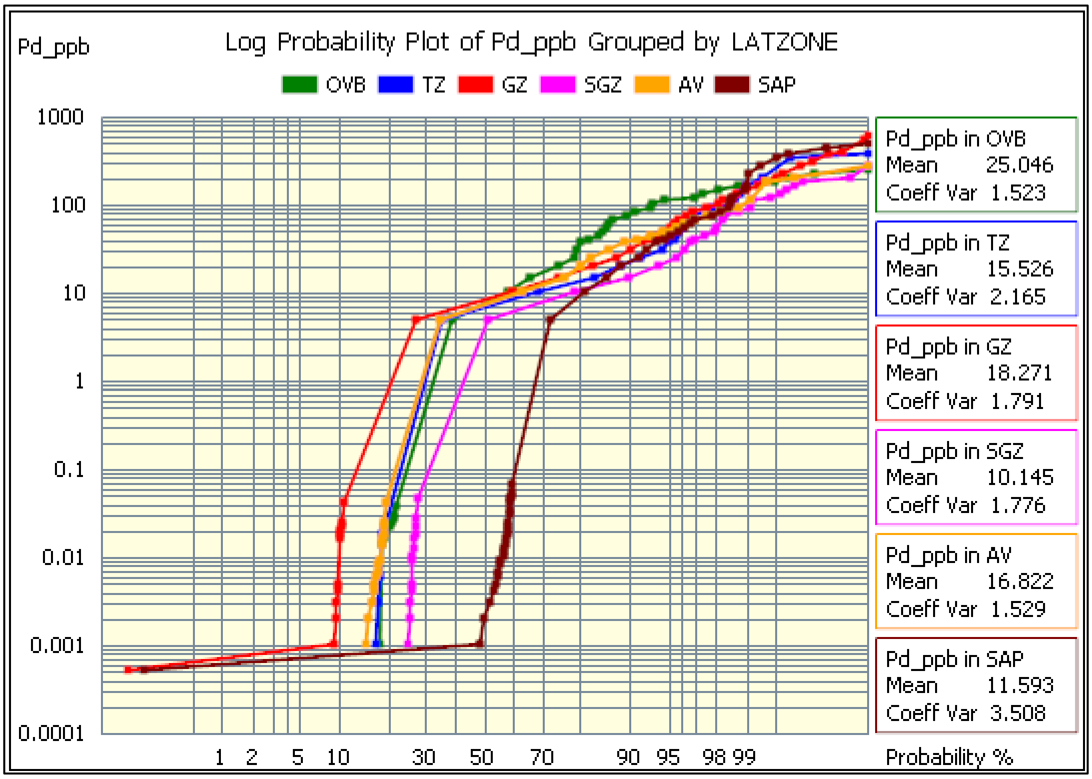



**Figure 14-29: Log probability plot of lead**

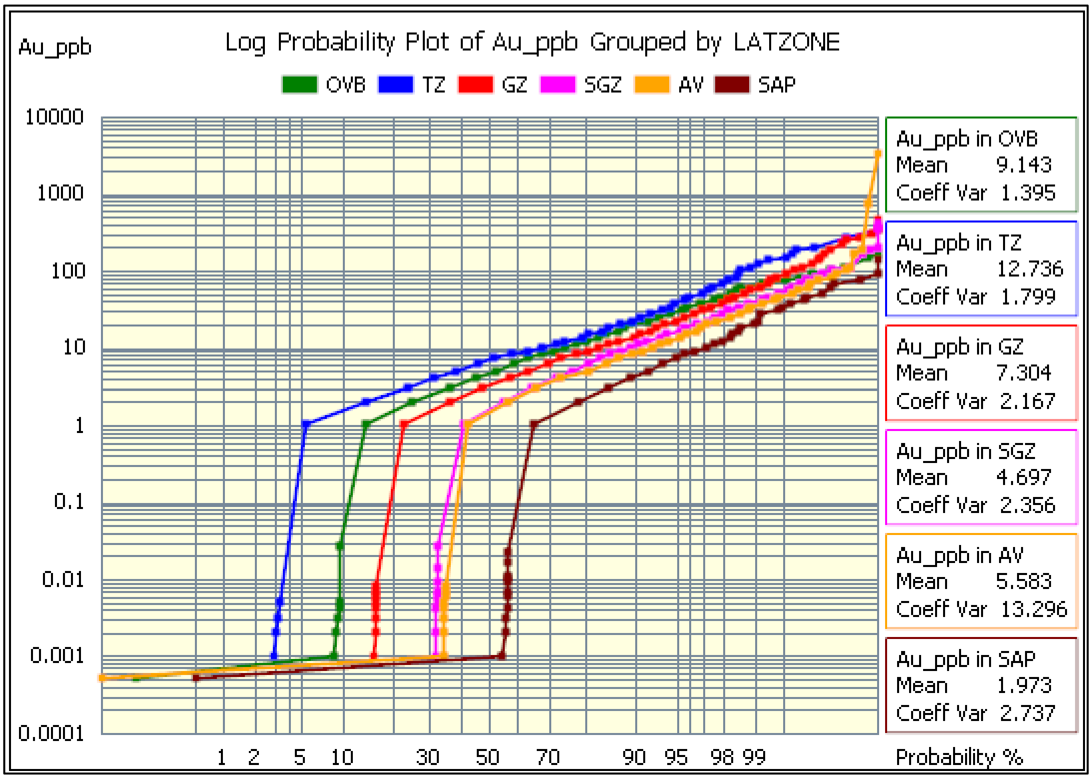



**Figure 14-30: Log probability plot of gold**

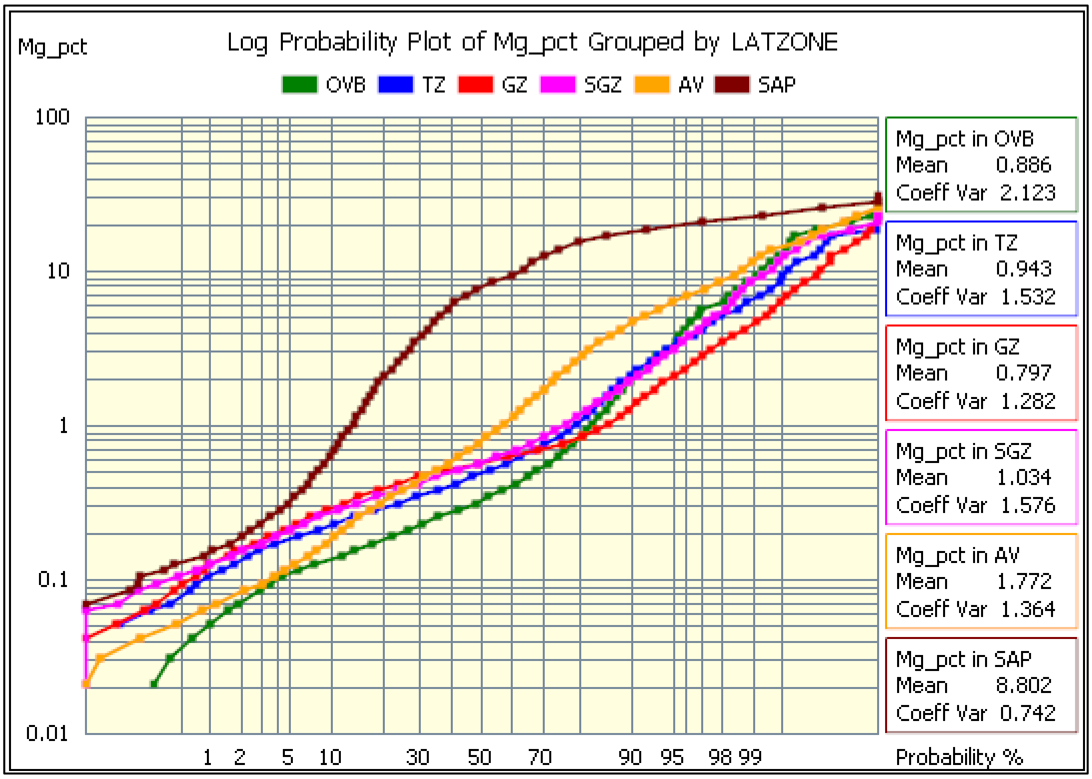



**Figure 14-31: Log probability plot of magnesium**

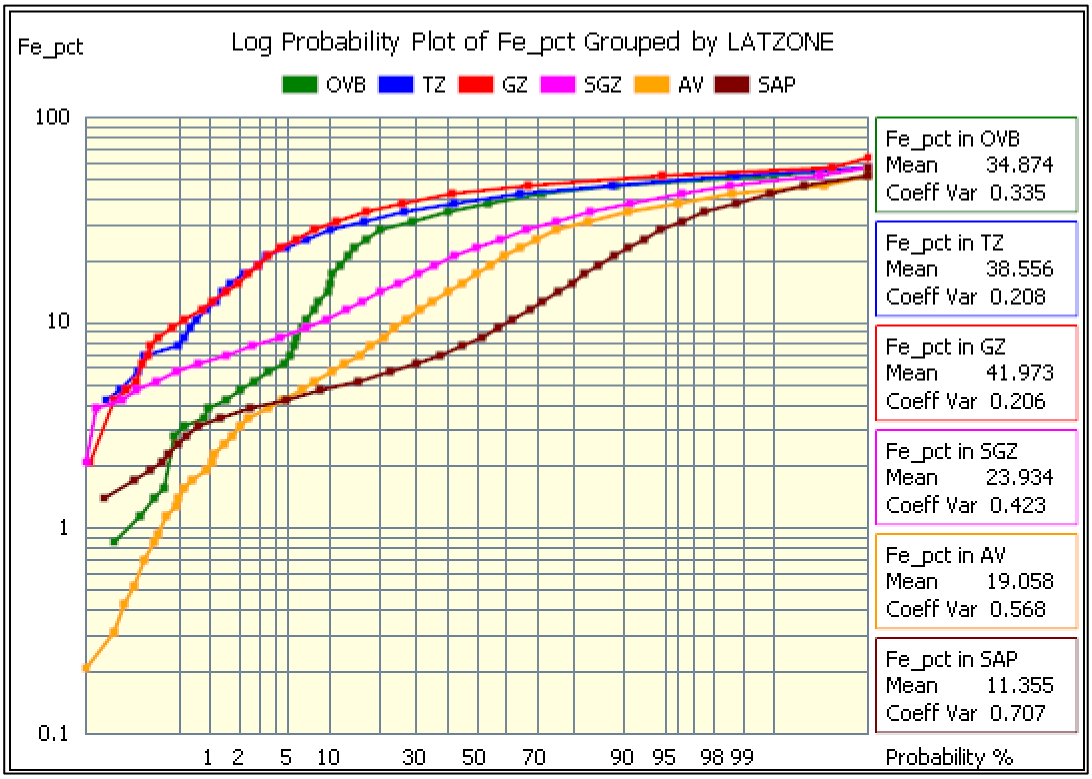



**Figure 14-32: Log probability plot of iron**

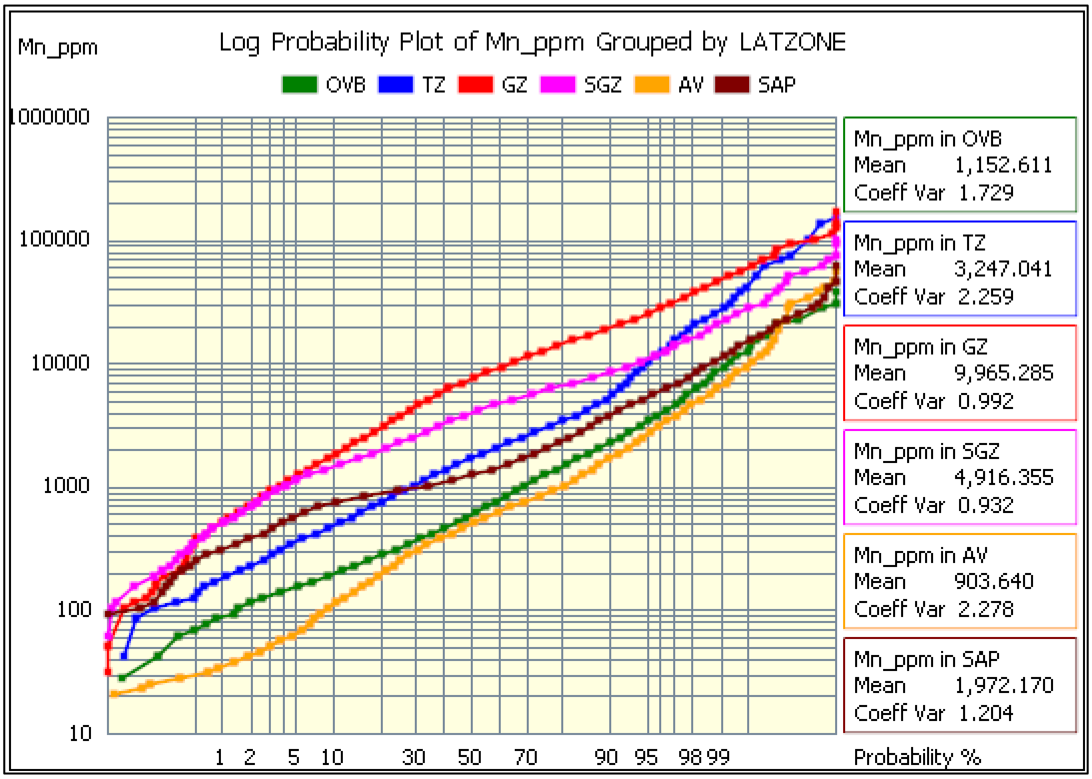



**Figure 14-33: Log probability plot of manganese**

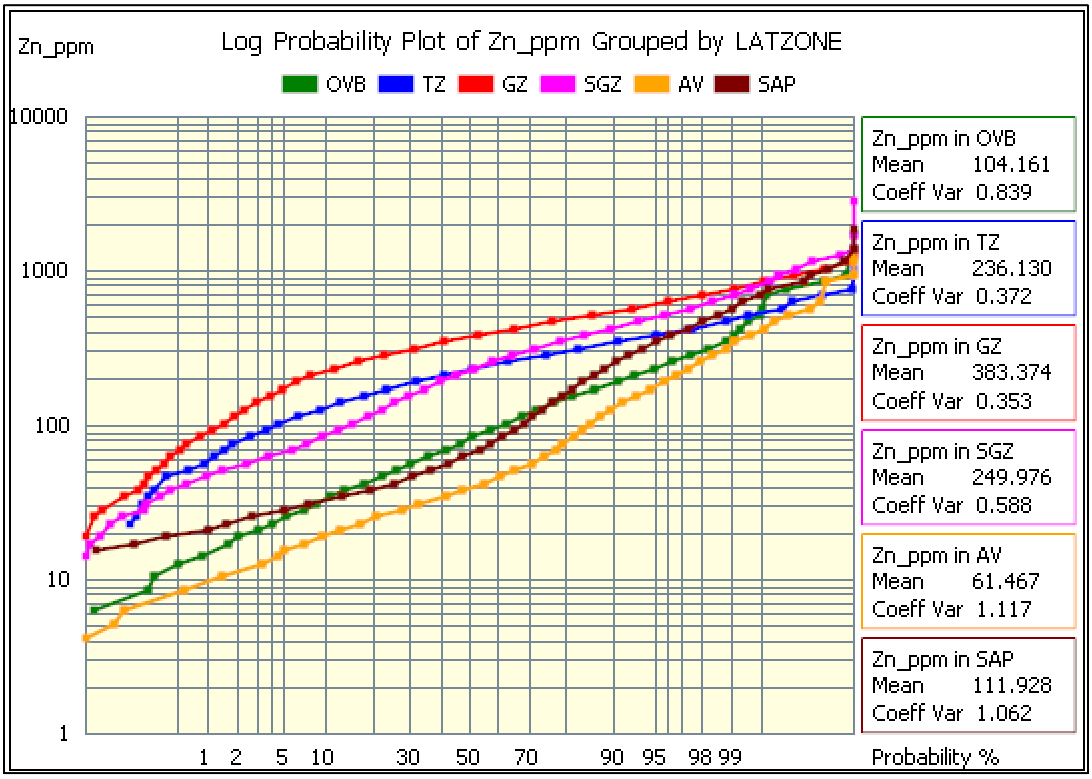



**Figure 14-34: Log probability plot of zinc**

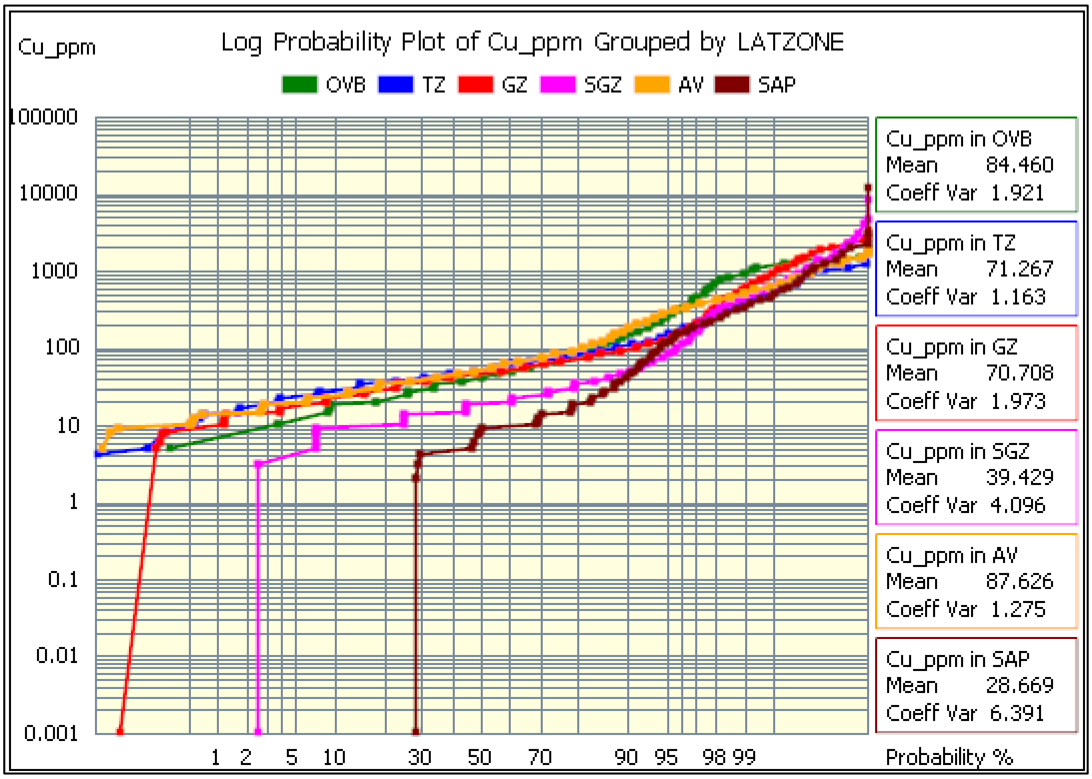



**Figure 14-35: Log probability plot of copper**

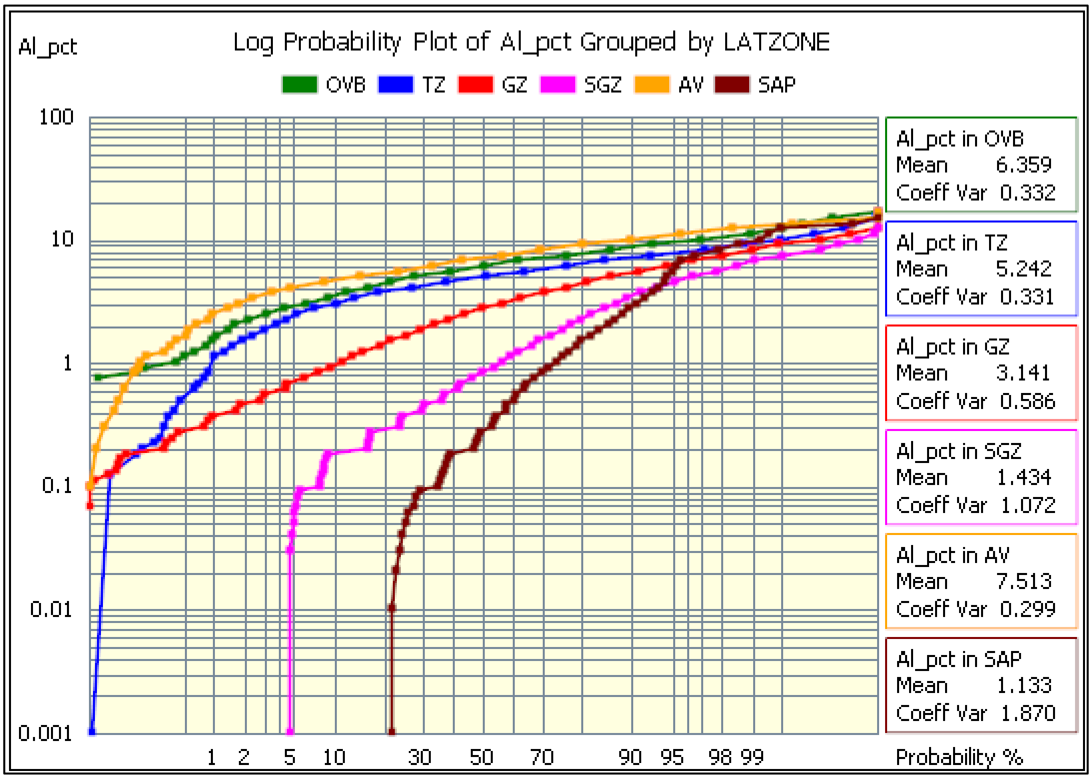



**Figure 14-36: Log probability plot of aluminium**

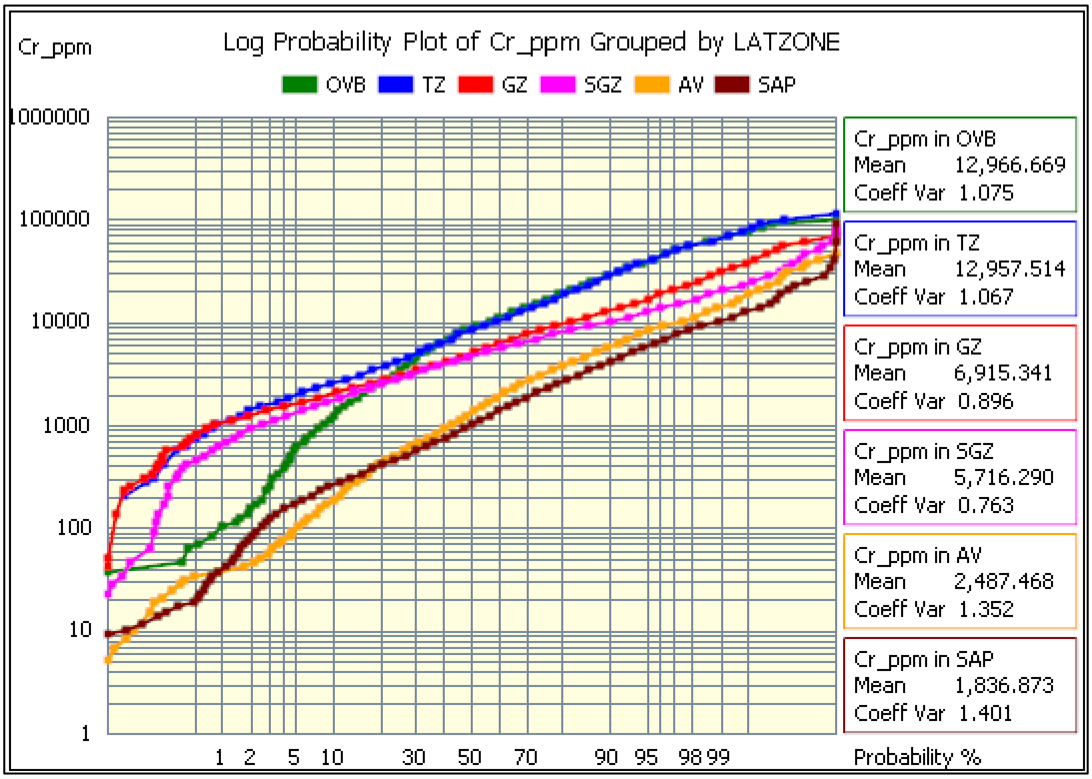



**Figure 14-37: Log probability plot of chromium**

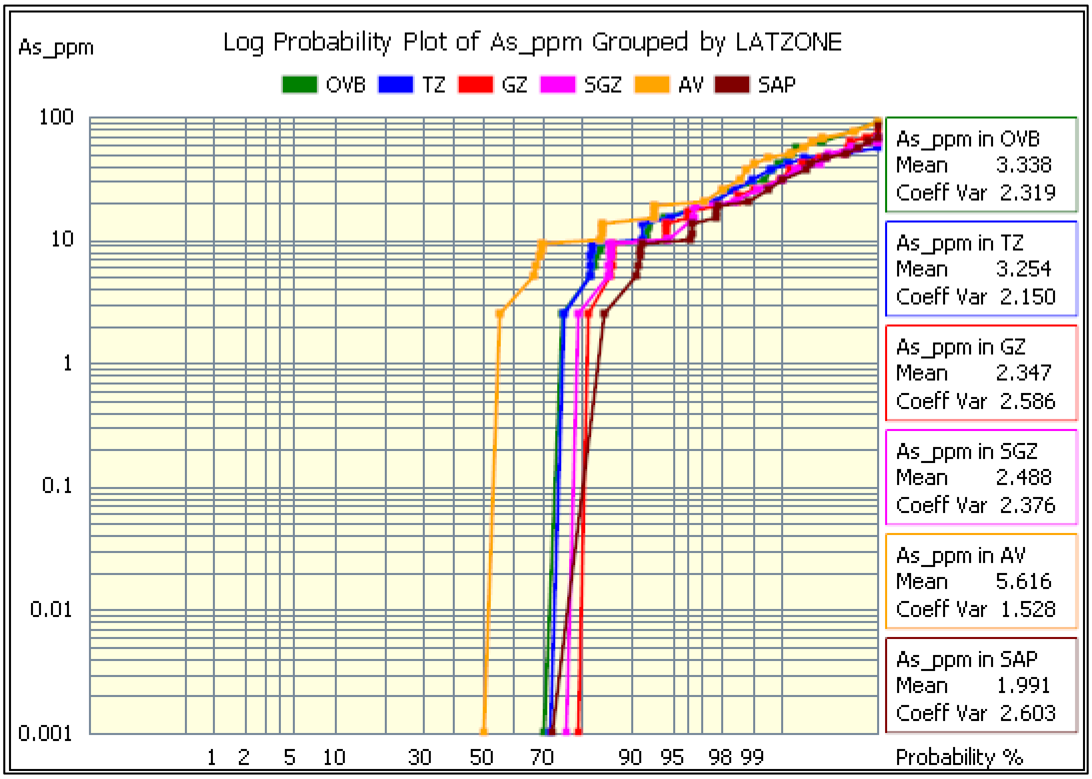



**Figure 14-38: Log probability plot of arsenic**

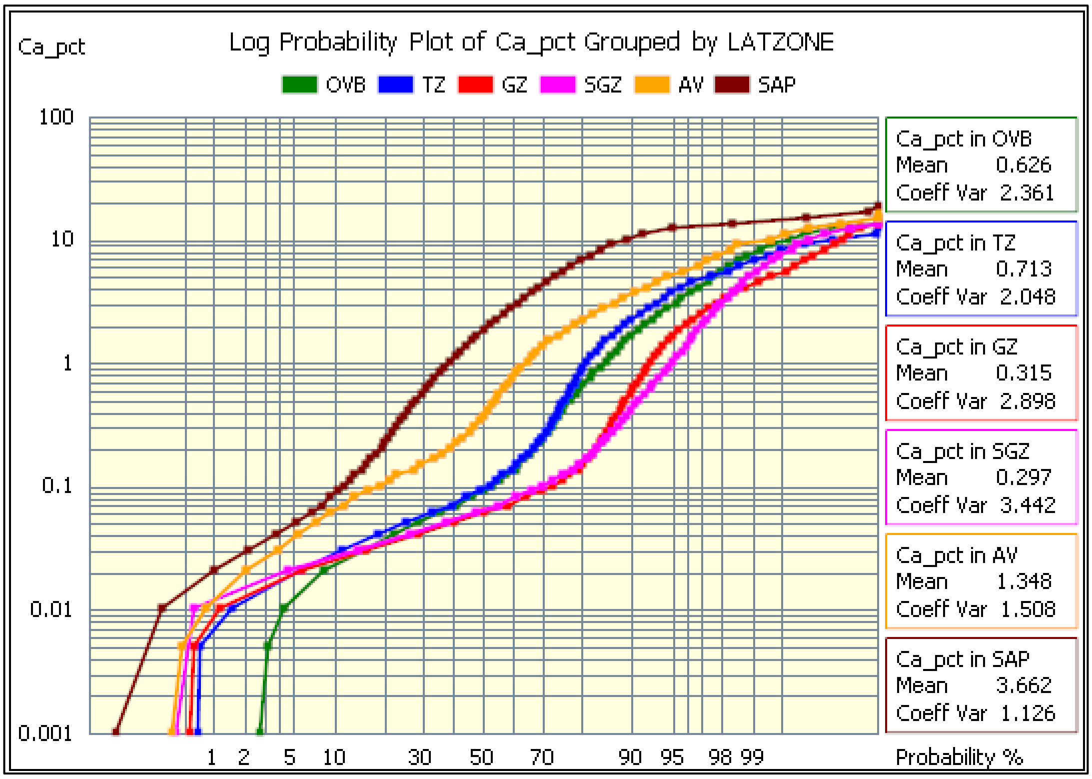



**Figure 14-39: Log probability plot of calcium**

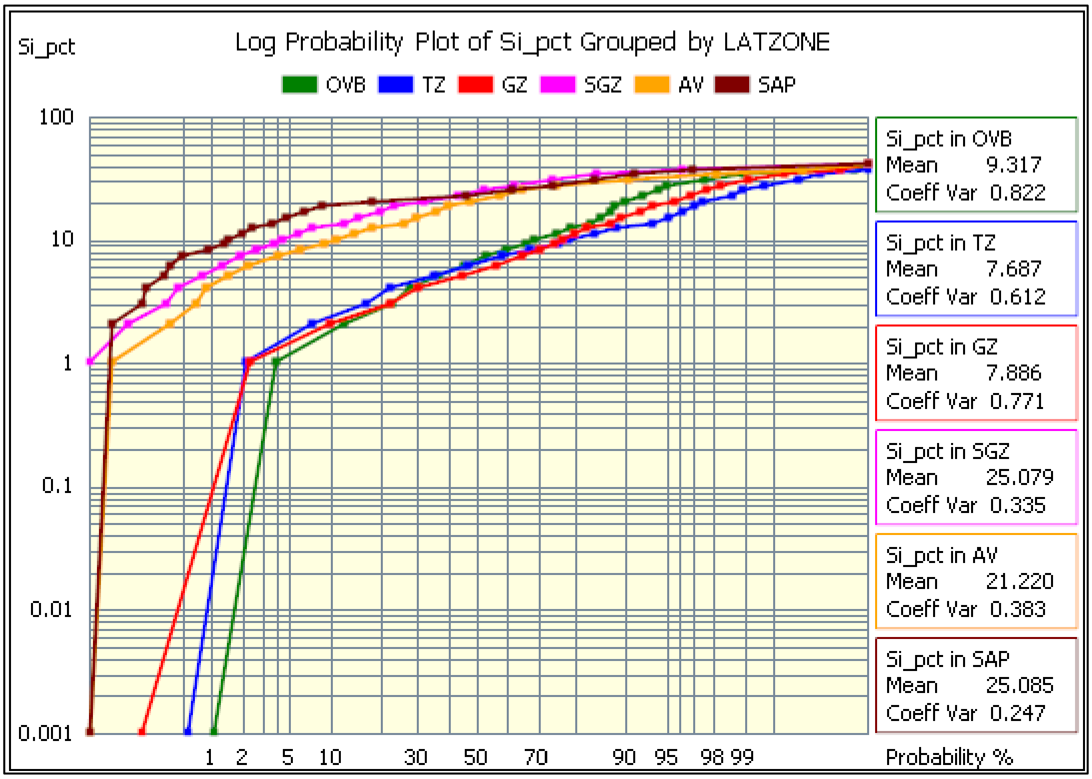



**Figure 14-40: Log probability plot of silica**

### 14.2.6 Domain boundary analysis

While it is clear from overlay plots of the distribution of all elements by LATZONE (see an example below), a boundary analysis (Figure 14-41 to Figure 14-48) has been carried out to confirm that hard rather than soft boundaries should be used in the estimation.

With the time constraints on the resource estimation, the interfaces between major domains (TZ, GZ and SGZ) have been reviewed for the major elements to be estimated (including Ni, Co, Sc, Pt, Fe, Si, Al and Mn). It has been concluded that hard wiring of boundaries is appropriate.

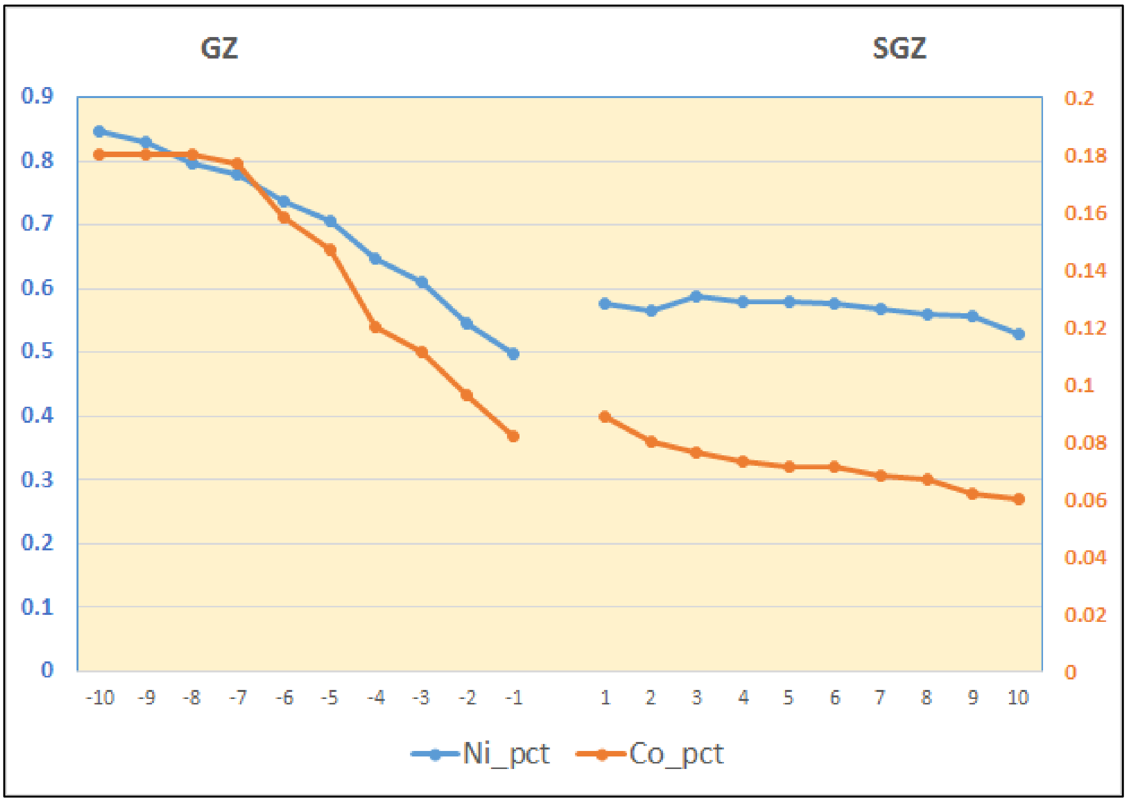



**Figure 14-41: Boundary analysis GZ/SGZ Ni and Co**

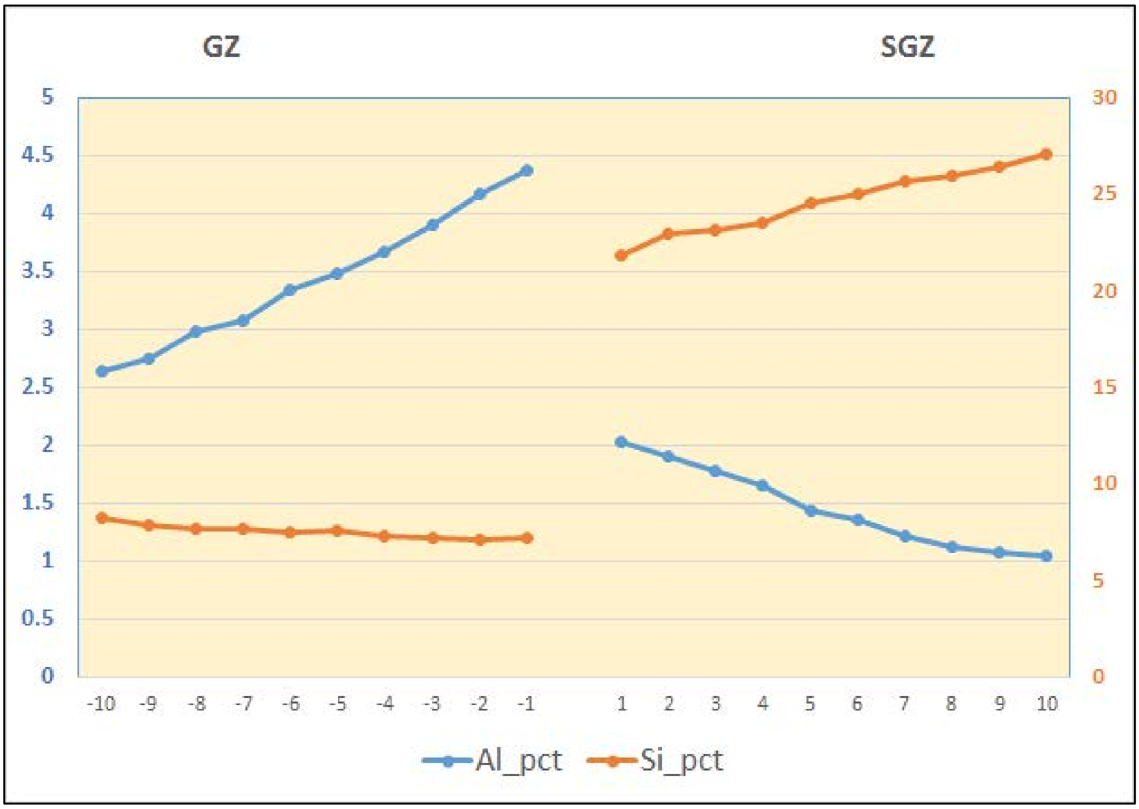



**Figure 14-42: Boundary analysis GZ/SGZ Al and Si**

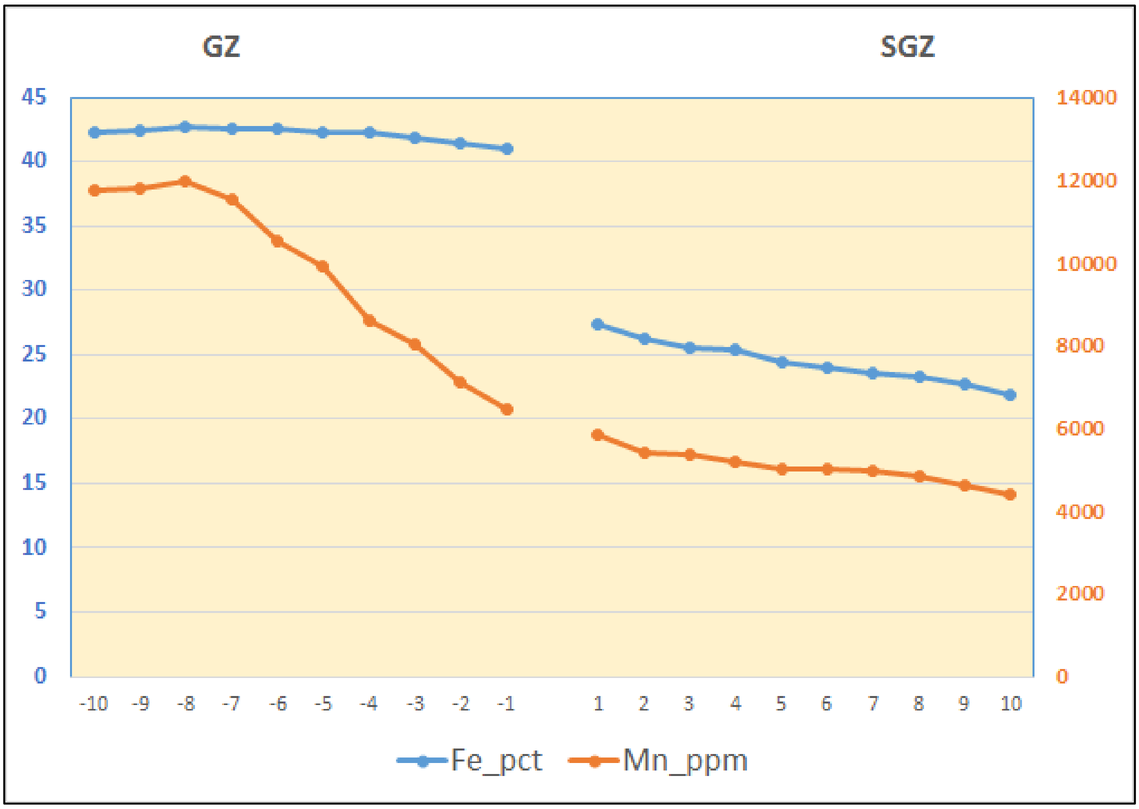



**Figure 14-43: Boundary analysis GZ/SGZ Fe and Mn**

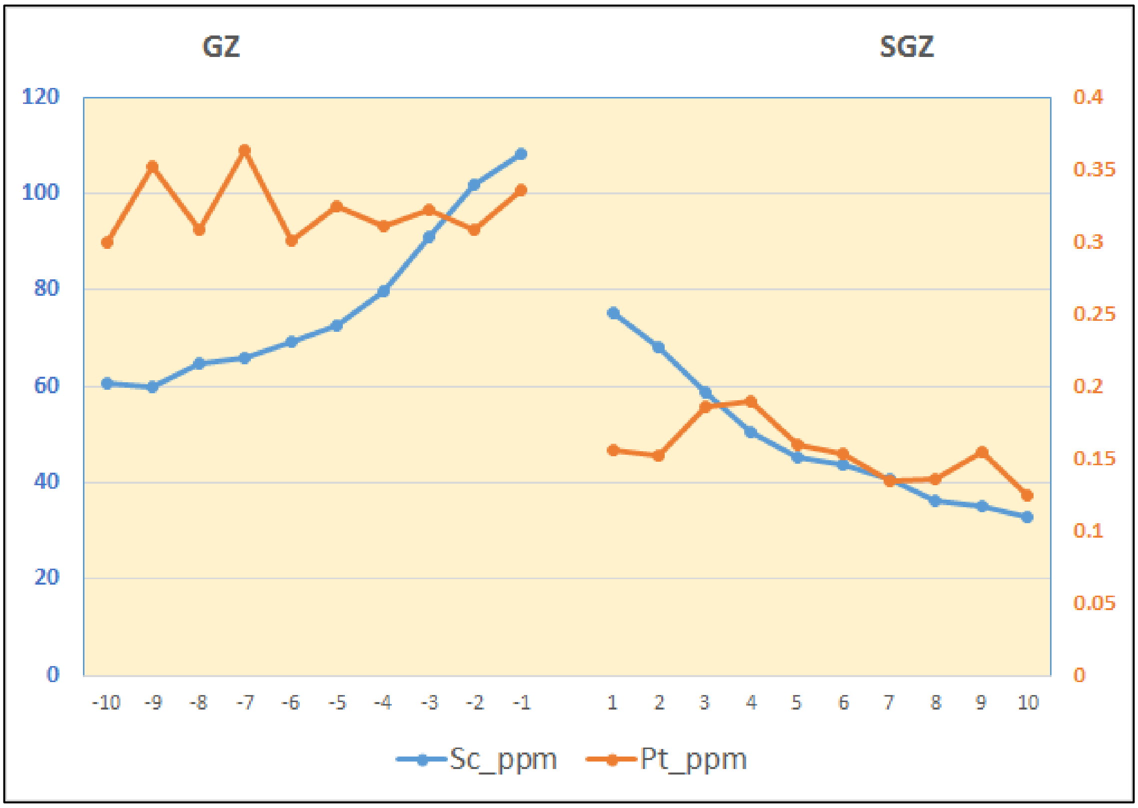



**Figure 14-44: Boundary analysis GZ/SGZ Sc and Pt**

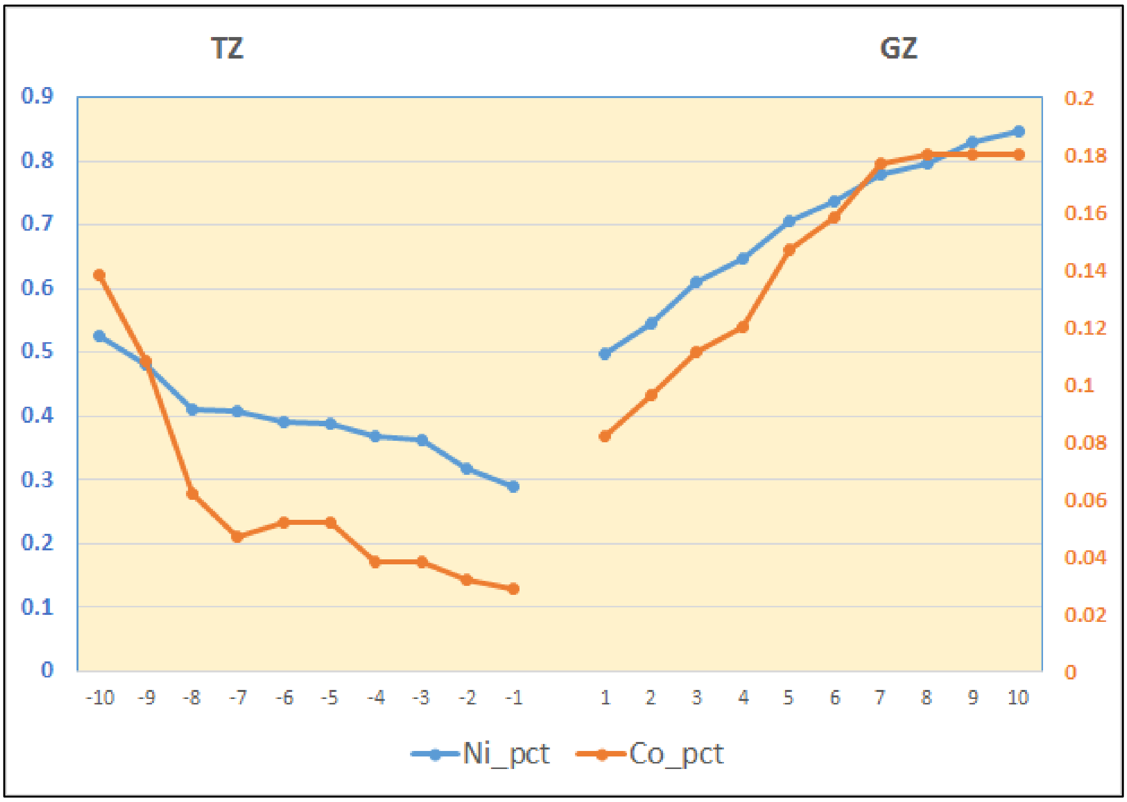



Figure 14-45: Boundary analysis TZ/GZ Ni and Co

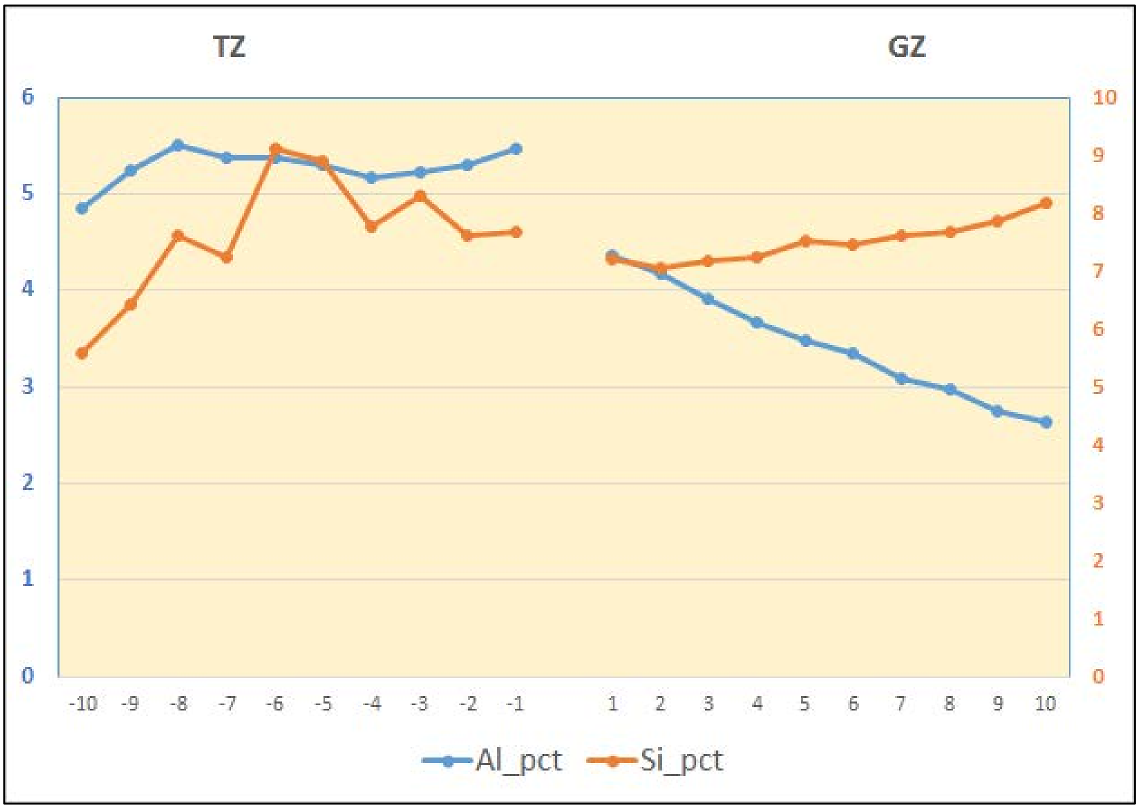



Figure 14-46: Boundary analysis TZ/GZ Al and Si

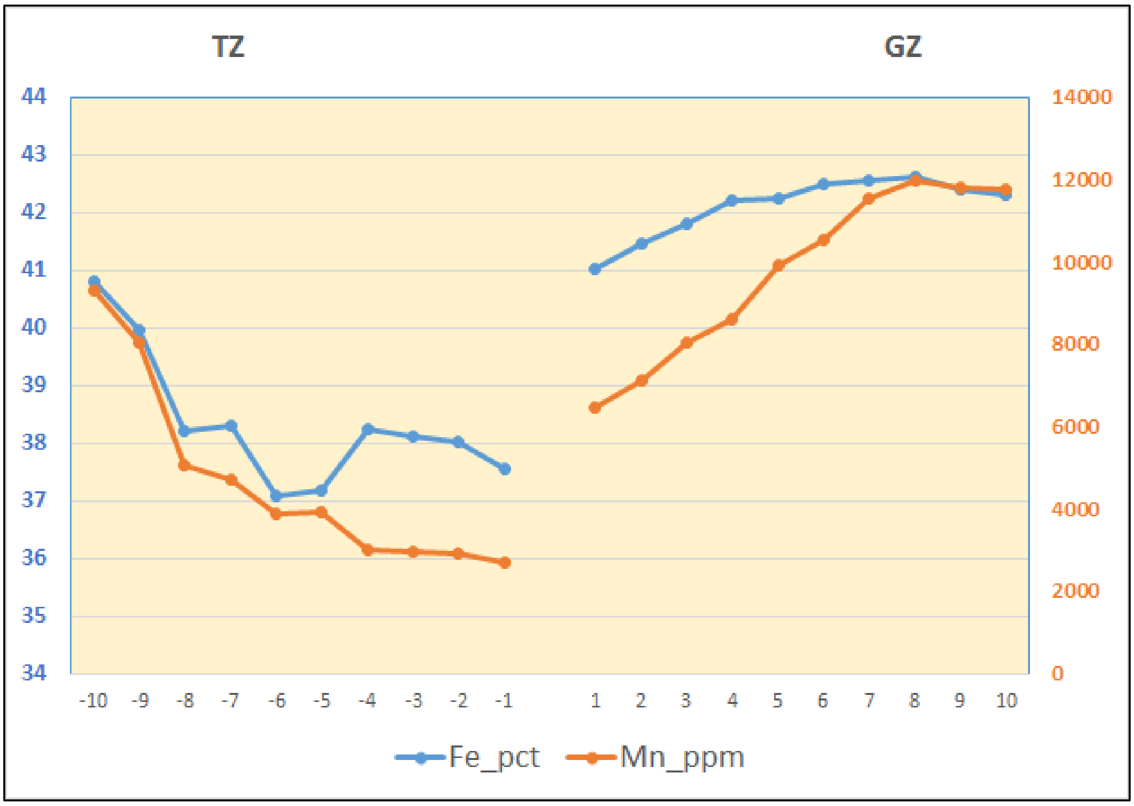



**Figure 14-47: Boundary analysis TZ/GZ Fe and Mn**

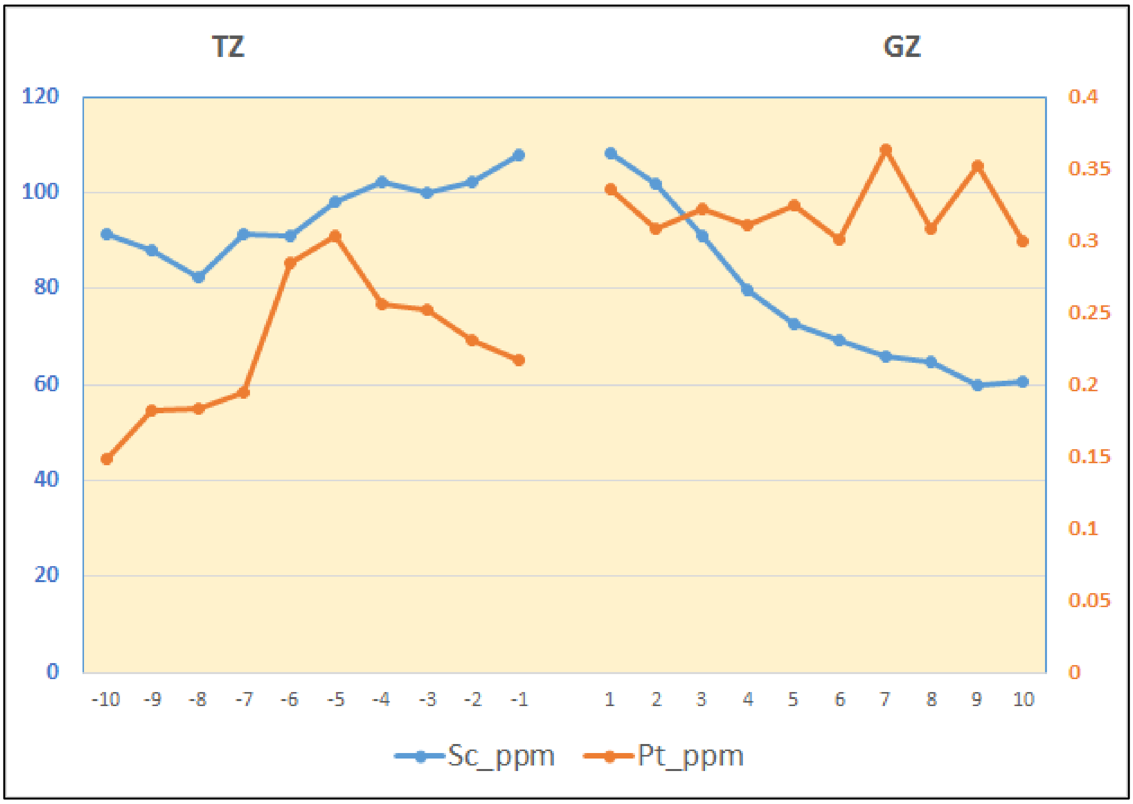



**Figure 14-48: Boundary analysis TZ/GZ Sc and Pt**

## 14.2.7 Correlation analysis

Correlation analysis was carried out on all elements by LATZONE, but showed rather poor correlation in general, apart from a few pairs of elements such as Pt/Pd, Fe/Si, Ca/Mg, CO/Mn. Correlation matrices are illustrated in Figure 14-49 to Figure 14-53.

| LATZONE AV | Au_ppb | Pt_ppm | Pd_ppb | Ni_pct | Co_pct | Mg_pct | Fe_pct | Mn_ppm | Zn_ppm | Cu_ppm | Al_pct | Cr_ppm | As_ppm | Ca_pct | Sc_ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pt_ppm | 0.00 | | | | | | | | | | | | | | |
| Pd_ppb | 0.11 | 0.07 | | | | | | | | | | | | | |
| Ni_pct | 0.00 | 0.26 | -0.11 | | | | | | | | | | | | |
| Co_pct | 0.00 | 0.12 | 0.24 | 0.52 | | | | | | | | | | | |
| Mg_pct | -0.02 | 0.03 | -0.14 | 0.14 | 0.04 | | | | | | | | | | |
| Fe_pct | 0.01 | 0.23 | 0.23 | 0.37 | 0.20 | -0.30 | | | | | | | | | |
| Mn_ppm | 0.00 | 0.12 | 0.19 | 0.45 | 0.85 | 0.04 | 0.22 | | | | | | | | |
| Zn_ppm | 0.00 | 0.20 | 0.11 | 0.61 | 0.49 | 0.02 | 0.34 | 0.47 | | | | | | | |
| Cu_ppm | 0.02 | -0.01 | 0.42 | 0.04 | 0.36 | -0.10 | 0.17 | 0.46 | 0.25 | | | | | | |
| Al_pct | -0.01 | -0.11 | 0.03 | -0.27 | -0.12 | -0.25 | -0.03 | -0.08 | -0.23 | 0.14 | | | | | |
| Cr_ppm | 0.00 | 0.15 | -0.17 | 0.28 | 0.08 | -0.14 | 0.53 | 0.03 | 0.18 | -0.10 | -0.09 | | | | |
| As_ppm | -0.01 | -0.08 | -0.16 | -0.17 | -0.10 | -0.06 | -0.26 | -0.10 | -0.14 | -0.11 | -0.10 | -0.20 | | | |
| Ca_pct | -0.02 | 0.00 | -0.11 | 0.03 | -0.02 | 0.65 | -0.31 | -0.02 | -0.01 | -0.09 | -0.21 | -0.16 | 0.00 | | |
| Sc_ppm | 0.01 | 0.09 | 0.47 | 0.05 | 0.17 | -0.24 | 0.57 | 0.18 | 0.30 | 0.26 | 0.10 | 0.17 | -0.18 | -0.19 | |
| Si_pct | 0.00 | -0.19 | -0.36 | -0.42 | -0.21 | -0.08 | -0.91 | -0.23 | -0.31 | -0.16 | -0.20 | -0.51 | 0.37 | -0.03 | -0.54 |

**Figure 14-49: Correlation matrix AV**

| LATZONE OVB | Au_ppb | Pt_ppm | Pd_ppb | Ni_pct | Co_pct | Mg_pct | Fe_pct | Mn_ppm | Zn_ppm | Cu_ppm | Al_pct | Cr_ppm | As_ppm | Ca_pct | Sc_ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pt_ppm | -0.03 | | | | | | | | | | | | | | |
| Pd_ppb | 0.35 | -0.02 | | | | | | | | | | | | | |
| Ni_pct | 0.10 | 0.05 | -0.23 | | | | | | | | | | | | |
| Co_pct | 0.02 | 0.04 | -0.04 | 0.32 | | | | | | | | | | | |
| Mg_pct | -0.09 | -0.02 | -0.07 | 0.08 | 0.01 | | | | | | | | | | |
| Fe_pct | 0.09 | 0.03 | -0.05 | 0.16 | 0.00 | -0.39 | | | | | | | | | |
| Mn_ppm | 0.02 | 0.06 | 0.16 | 0.28 | 0.76 | 0.03 | 0.00 | | | | | | | | |
| Zn_ppm | 0.12 | 0.02 | -0.02 | 0.46 | 0.38 | 0.04 | -0.01 | 0.36 | | | | | | | |
| Cu_ppm | 0.32 | -0.02 | 0.50 | -0.11 | 0.08 | -0.02 | -0.12 | 0.19 | 0.24 | | | | | | |
| Al_pct | 0.01 | -0.05 | 0.22 | -0.35 | -0.10 | -0.24 | -0.32 | -0.04 | -0.18 | 0.15 | | | | | |
| Cr_ppm | 0.08 | 0.01 | -0.35 | 0.16 | -0.06 | -0.15 | 0.34 | -0.22 | 0.01 | -0.18 | -0.32 | | | | |
| As_ppm | -0.05 | -0.02 | -0.14 | -0.05 | -0.02 | -0.08 | -0.20 | -0.01 | 0.07 | 0.06 | 0.10 | -0.08 | | | |
| Ca_pct | -0.04 | -0.03 | 0.00 | 0.02 | 0.04 | 0.56 | -0.41 | 0.03 | 0.00 | 0.01 | -0.15 | -0.19 | -0.07 | | |
| Sc_ppm | 0.11 | -0.02 | 0.49 | -0.15 | 0.17 | -0.17 | 0.05 | 0.28 | 0.14 | 0.28 | 0.33 | -0.28 | -0.08 | -0.10 | |
| Si_pct | -0.10 | -0.03 | -0.30 | -0.13 | -0.02 | 0.27 | -0.95 | 0.00 | 0.01 | 0.11 | 0.23 | -0.43 | 0.31 | 0.31 | -0.13 |

**Figure 14-50: Correlation matrix OVB**

| LATZONE TZ | Au_ppb | Pt_ppm | Pd_ppb | Ni_pct | Co_pct | Mg_pct | Fe_pct | Mn_ppm | Zn_ppm | Cu_ppm | Al_pct | Cr_ppm | As_ppm | Ca_pct | Sc_ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pt_ppm | -0.03 | | | | | | | | | | | | | | |
| Pd_ppb | 0.33 | 0.28 | | | | | | | | | | | | | |
| Ni_pct | 0.15 | 0.07 | 0.00 | | | | | | | | | | | | |
| Co_pct | 0.26 | 0.15 | 0.37 | 0.34 | | | | | | | | | | | |
| Mg_pct | -0.08 | 0.01 | -0.09 | -0.07 | -0.08 | | | | | | | | | | |
| Fe_pct | 0.08 | 0.12 | -0.20 | 0.24 | 0.07 | -0.42 | | | | | | | | | |
| Mn_ppm | 0.17 | 0.23 | 0.26 | 0.20 | 0.76 | -0.03 | 0.02 | | | | | | | | |
| Zn_ppm | 0.06 | 0.08 | 0.09 | 0.66 | 0.37 | -0.16 | 0.33 | 0.20 | | | | | | | |
| Cu_ppm | 0.19 | -0.05 | 0.27 | -0.06 | 0.08 | -0.09 | -0.01 | 0.06 | 0.09 | | | | | | |
| Al_pct | 0.05 | -0.07 | 0.07 | -0.22 | -0.06 | -0.27 | -0.25 | -0.04 | -0.22 | 0.17 | | | | | |
| Cr_ppm | -0.03 | -0.06 | -0.13 | -0.03 | -0.14 | -0.21 | 0.07 | -0.17 | 0.02 | -0.05 | 0.12 | | | | |
| As_ppm | -0.02 | -0.01 | 0.00 | 0.05 | 0.04 | -0.08 | -0.01 | 0.00 | 0.06 | -0.02 | 0.04 | 0.12 | | | |
| Ca_pct | -0.07 | 0.05 | -0.09 | -0.07 | -0.09 | 0.61 | -0.38 | -0.03 | -0.17 | -0.10 | -0.21 | -0.24 | -0.09 | | |
| Sc_ppm | 0.06 | -0.02 | 0.13 | -0.27 | 0.07 | -0.17 | 0.01 | 0.04 | -0.04 | 0.34 | 0.29 | -0.08 | 0.01 | -0.13 | |
| Si_pct | -0.08 | -0.14 | -0.05 | -0.17 | -0.09 | 0.25 | -0.91 | -0.07 | -0.26 | 0.01 | 0.14 | -0.20 | 0.00 | 0.24 | -0.10 |

**Figure 14-51: Correlation matrix TZ**

| LATZONE GZ | Au_ppb | Pt_ppm | Pd_ppb | Ni_pct | Co_pct | Mg_pct | Fe_pct | Mn_ppm | Zn_ppm | Cu_ppm | Al_pct | Cr_ppm | As_ppm | Ca_pct | Sc_ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pt_ppm | 0.06 | | | | | | | | | | | | | | |
| Pd_ppb | 0.41 | 0.43 | | | | | | | | | | | | | |
| Ni_pct | -0.12 | 0.00 | -0.07 | | | | | | | | | | | | |
| Co_pct | -0.02 | -0.01 | 0.10 | 0.36 | | | | | | | | | | | |
| Mg_pct | -0.05 | 0.00 | -0.05 | 0.01 | -0.07 | | | | | | | | | | |
| Fe_pct | -0.05 | 0.02 | -0.07 | 0.24 | 0.06 | -0.31 | | | | | | | | | |
| Mn_ppm | -0.03 | 0.00 | 0.10 | 0.28 | 0.78 | -0.07 | 0.09 | | | | | | | | |
| Zn_ppm | -0.06 | 0.00 | -0.02 | 0.57 | 0.46 | -0.11 | 0.40 | 0.35 | | | | | | | |
| Cu_ppm | 0.19 | -0.02 | 0.41 | -0.11 | 0.08 | -0.01 | 0.01 | 0.10 | 0.05 | | | | | | |
| Al_pct | 0.13 | 0.00 | 0.21 | -0.22 | 0.09 | -0.04 | -0.19 | 0.04 | -0.23 | 0.20 | | | | | |
| Cr_ppm | 0.01 | 0.10 | 0.23 | 0.08 | -0.16 | -0.09 | 0.01 | -0.20 | -0.01 | -0.06 | 0.04 | | | | |
| As_ppm | 0.04 | 0.00 | -0.03 | -0.05 | -0.02 | -0.01 | -0.01 | -0.02 | -0.03 | -0.01 | 0.02 | 0.06 | | | |
| Ca_pct | 0.00 | -0.01 | -0.05 | -0.21 | -0.10 | 0.61 | -0.27 | -0.09 | -0.21 | -0.01 | 0.02 | -0.12 | -0.01 | | |
| Sc_ppm | 0.09 | -0.01 | 0.19 | -0.23 | 0.01 | -0.06 | -0.12 | 0.00 | -0.15 | 0.15 | 0.44 | -0.04 | 0.02 | 0.01 | |
| SI_pct | 0.01 | -0.02 | 0.03 | -0.17 | -0.13 | 0.16 | -0.92 | -0.16 | -0.31 | -0.07 | -0.10 | -0.03 | -0.02 | 0.11 | -0.01 |

**Figure 14-52: Correlation matrix GZ**

| LATZONE SGZ | Au_ppb | Pt_ppm | Pd_ppb | Ni_pct | Co_pct | Mg_pct | Fe_pct | Mn_ppm | Zn_ppm | Cu_ppm | Al_pct | Cr_ppm | As_ppm | Ca_pct | Sc_ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pt_ppm | 0.00 | | | | | | | | | | | | | | |
| Pd_ppb | 0.53 | 0.12 | | | | | | | | | | | | | |
| Ni_pct | 0.00 | 0.09 | -0.07 | | | | | | | | | | | | |
| Co_pct | 0.09 | 0.06 | 0.17 | 0.41 | | | | | | | | | | | |
| Mg_pct | -0.03 | 0.01 | 0.05 | 0.11 | -0.08 | | | | | | | | | | |
| Fe_pct | 0.09 | 0.12 | 0.11 | 0.61 | 0.45 | -0.13 | | | | | | | | | |
| Mn_ppm | 0.05 | 0.07 | 0.16 | 0.39 | 0.72 | -0.05 | 0.43 | | | | | | | | |
| Zn_ppm | 0.14 | 0.08 | 0.18 | 0.66 | 0.53 | -0.05 | 0.71 | 0.42 | | | | | | | |
| Cu_ppm | 0.30 | -0.01 | 0.39 | -0.08 | 0.09 | -0.01 | 0.11 | 0.09 | 0.18 | | | | | | |
| Al_pct | 0.30 | 0.02 | 0.35 | -0.04 | 0.19 | -0.02 | 0.22 | 0.13 | 0.19 | 0.27 | | | | | |
| Cr_ppm | -0.04 | 0.15 | -0.10 | 0.45 | 0.13 | -0.07 | 0.44 | 0.15 | 0.37 | -0.04 | 0.01 | | | | |
| As_ppm | 0.06 | 0.02 | 0.03 | 0.04 | 0.10 | -0.01 | 0.09 | 0.05 | 0.09 | 0.03 | 0.06 | 0.07 | | | |
| Ca_pct | -0.02 | -0.02 | 0.01 | -0.13 | -0.05 | 0.51 | -0.19 | -0.08 | -0.10 | 0.02 | 0.02 | -0.14 | 0.00 | | |
| Sc_ppm | 0.13 | 0.04 | 0.22 | -0.07 | 0.15 | 0.04 | 0.23 | 0.18 | 0.19 | 0.29 | 0.62 | -0.04 | 0.02 | 0.11 | |
| SI_pct | -0.16 | -0.11 | -0.41 | -0.67 | -0.48 | -0.13 | -0.95 | -0.42 | -0.74 | -0.09 | -0.36 | -0.47 | -0.11 | 0.00 | -0.31 |

**Figure 14-53: Correlation matrix SGZ**

## 14.2.8 Variography

Variography has been carried out for the GZ, SGZ and TZ zones for Co, Ni, Fe, Si, Al, Mg, Mn, Pt and Sc. TZ variograms have been assigned to the AV and OVB domains, and SGZ variograms to the SAP domain. A summary of variogram parameters is shown below, and modelled variograms are illustrated in Table 14-14 to Table 14-16 and Figure 14-54 to Figure 14-80.

**Table 14-14: GZ variogram model parameters**

| LATZONE GZ | Variances | | | Major Axis (E-W) | | SemiMajor Axis (N-S) | | Minor Axis (vertical) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Variable | Nugget | Sill1 | Sill2 | Range1 | Range2 | Range1 | Range2 | Range1 | Range2 | Rotation |
| Ni_pct | 0.10 | 0.32 | 0.68 | 138.0 | 465.0 | 130.0 | 420.0 | 8.6 | 12.7 | 45.0 |
| Co_Pct | 0.10 | 0.42 | 0.48 | 109.0 | 198.0 | 50.0 | 190.0 | 2.9 | 5.0 | 10.0 |
| Fe_ptc | 0.12 | 0.42 | 0.31 | 89.0 | 225.0 | 50.0 | 230.0 | 4.6 | 15.8 | 0.0 |
| Si_pct | 0.08 | 0.44 | 0.37 | 119.0 | 257.0 | 100.0 | 225.0 | 7.7 | 7.7 | 0.0 |
| Al_pct | 0.07 | 0.37 | 0.56 | 89.0 | 366.0 | 40.0 | 460.0 | 8.6 | 9.2 | 0.0 |
| Mg_pct | 0.08 | 0.21 | 0.71 | 59.0 | 267.0 | 128.6 | 320.0 | 2.0 | 8.1 | 0.0 |
| Mn_ppm | 0.30 | 0.50 | 0.20 | 128.6 | 225.0 | 80.0 | 225.0 | 4.0 | 6.8 | 5.0 |
| Pt_ppm | 0.07 | 0.73 | 0.20 | 175.0 | 212.0 | 187.0 | 210.0 | 1.1 | 2.6 | 0.0 |
| Sc_ppm | 0.08 | 0.17 | 0.75 | 40.0 | 180.0 | 112.5 | 180.0 | 11.2 | 11.2 | 0.0 |

**Table 14-15: SGZ variogram model parameters**

| LATZONE SGZ | Variances | | | Major Axis (E-W) | | SemiMajor Axis (N-S) | | Minor Axis (vertical) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Variable | Nugget | Sill1 | Sill2 | Range1 | Range2 | Range1 | Range2 | Range1 | Range2 | Rotation |
| Ni_pct | 0.13 | 0.38 | 0.43 | 109.0 | 225.0 | 160.0 | 250.0 | 7.3 | 10.3 | 0.0 |
| Co_Pct | 0.12 | 0.30 | 0.58 | 128.6 | 225.0 | 100.0 | 370.0 | 2.6 | 6.6 | 10.0 |
| Fe_ptc | 0.12 | 0.37 | 0.43 | 89.0 | 225.0 | 80.0 | 225.0 | 3.7 | 5.5 | 0.0 |
| Si_pct | 0.08 | 0.36 | 0.46 | 128.6 | 158.0 | 40.0 | 160.0 | 2.9 | 5.0 | 0.0 |
| Al_pct | 0.03 | 0.46 | 0.38 | 148.0 | 702.0 | 170.0 | 800.0 | 7.5 | 10.8 | 0.0 |
| Mg_pct | 0.19 | 0.11 | 0.70 | 128.6 | 225.0 | 200.0 | 225.0 | 3.5 | 7.3 | 0.0 |
| Mn_ppm | 0.19 | 0.47 | 0.34 | 326.0 | 326.0 | 110.0 | 380.0 | 4.6 | 8.4 | 5.0 |
| Pt_ppm | 0.29 | 0.51 | 0.20 | 128.6 | 225.0 | 90.0 | 310.0 | 8.4 | 8.4 | 0.0 |
| Sc_ppm | 0.13 | 0.04 | 0.83 | 99.0 | 257.0 | 128.6 | 225.0 | 5.5 | 16.7 | 0.0 |

**Table 14-16: TZ variogram model parameters**

| LATZONE TZ | Variances | | | Major Axis (E-W) | | SemiMajor Axis (N-S) | | Minor Axis (vertical) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Variable | Nugget | Sill1 | Sill2 | Range1 | Range2 | Range1 | Range2 | Range1 | Range2 | Rotation |
| Ni_pct | 0.02 | 0.32 | 0.66 | 40.0 | 225.0 | 20.0 | 225.0 | 2.9 | 5.0 | 0.0 |
| Co_Pct | 0.10 | 0.70 | 0.20 | 128.6 | 225.0 | 50.0 | 225.0 | 1.8 | 5.0 | 0.0 |
| Fe_ptc | 0.05 | 0.07 | 0.77 | 49.0 | 188.0 | 60.0 | 225.0 | 2.9 | 6.6 | 0.0 |
| Si_pct | 0.24 | 0.06 | 0.70 | 128.6 | 267.0 | 110.0 | 290.0 | 5.3 | 10.3 | 0.0 |
| Al_pct | 0.15 | 0.47 | 0.38 | 79.0 | 225.0 | 80.0 | 225.0 | 4.4 | 5.0 | 0.0 |
| Mg_pct | 0.10 | 0.00 | 0.90 | 10.0 | 178.0 | 250.0 | 330.0 | 2.9 | 5.0 | 0.0 |
| Mn_ppm | 0.19 | 0.61 | 0.20 | 227.0 | 316.0 | 60.0 | 225.0 | 2.9 | 5.0 | 5.0 |
| Pt_ppm | 0.10 | 0.07 | 0.38 | 128.6 | 225.0 | 20.0 | 350.0 | 0.4 | 3.7 | 0.0 |
| Sc_ppm | 0.07 | 0.55 | 0.38 | 69.0 | 237.0 | 100.0 | 310.0 | 5.3 | 7.7 | 0.0 |

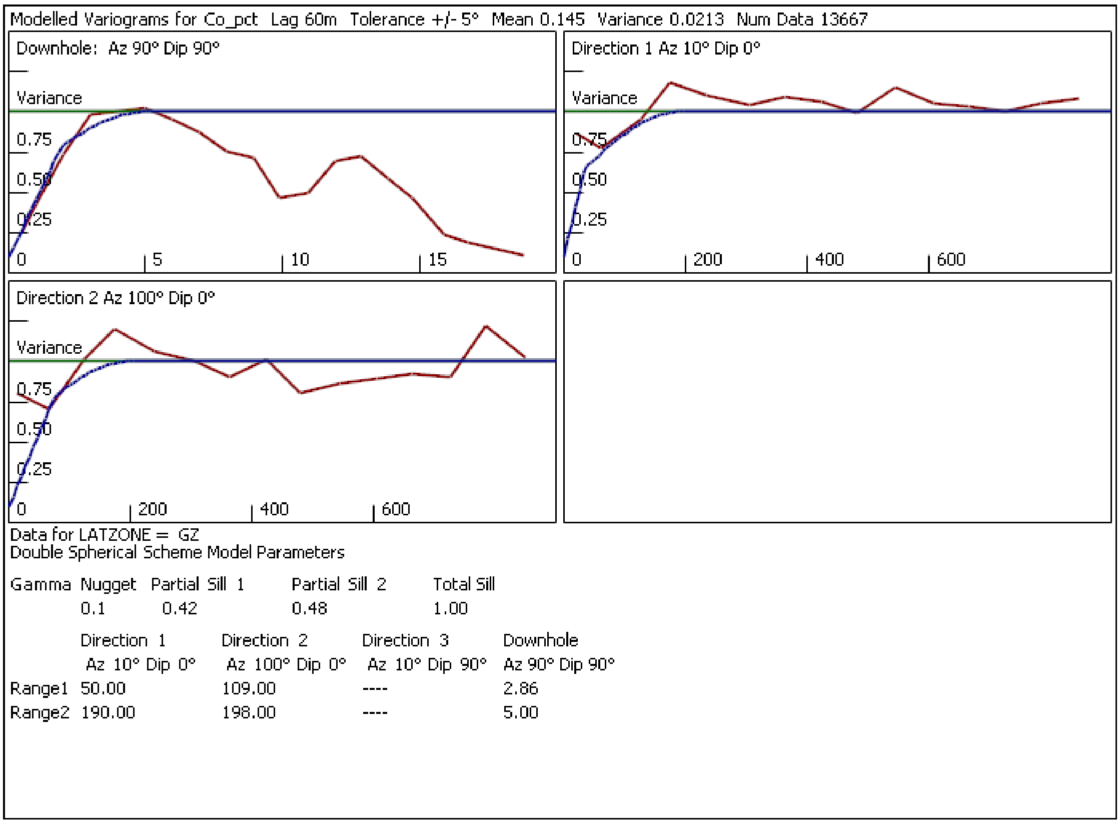



**Figure 14-54: GZ Co**

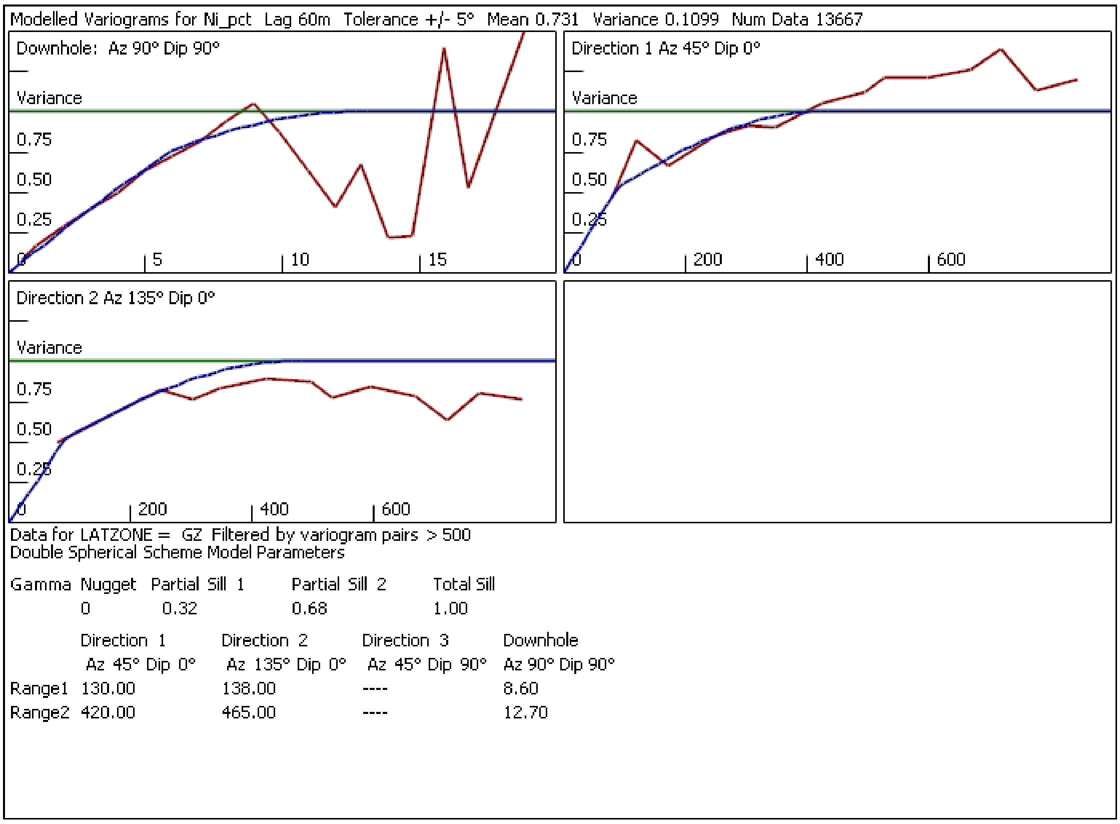



**Figure 14-55: GZ Ni**

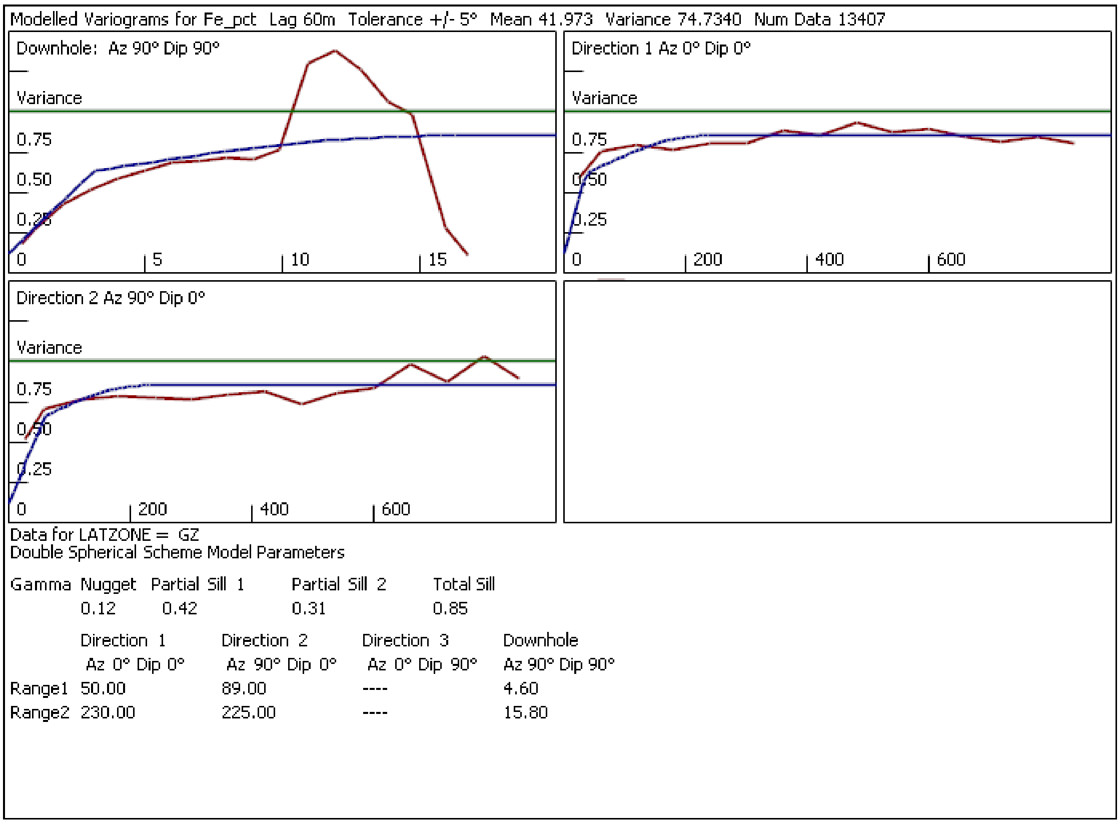



**Figure 14-56: GZ Fe**

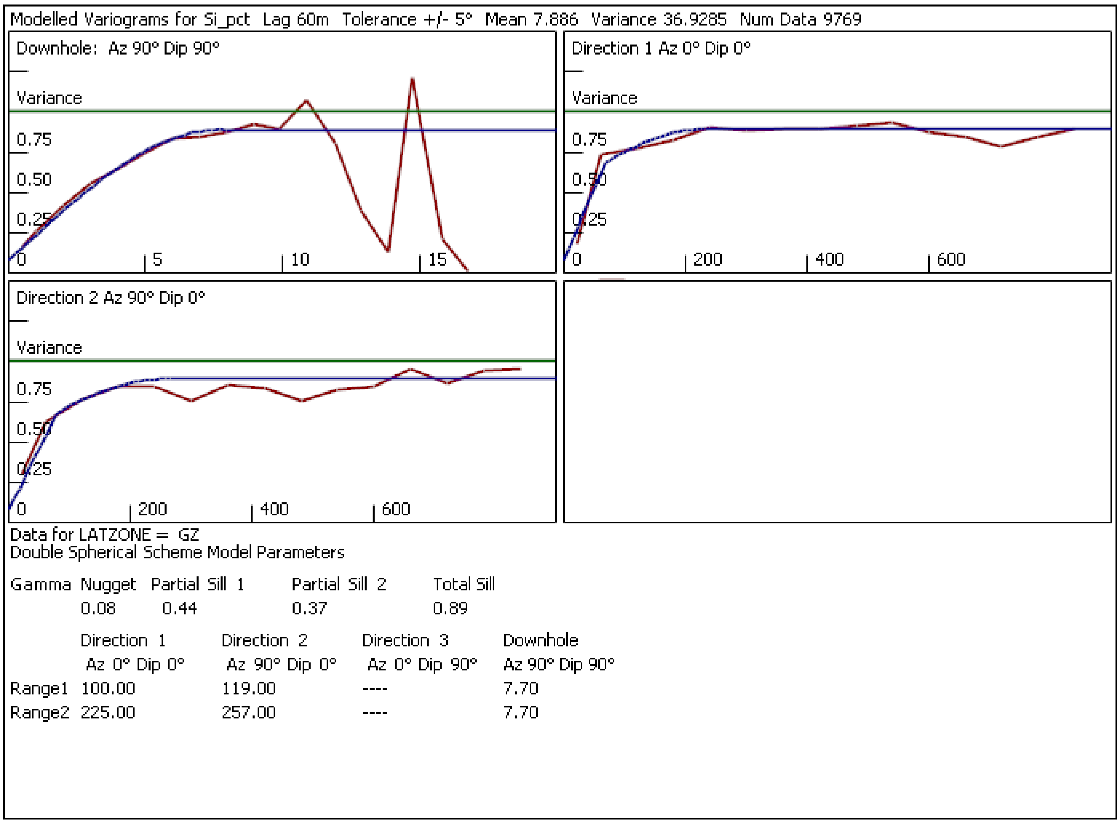



**Figure 14-57: GZ Si**

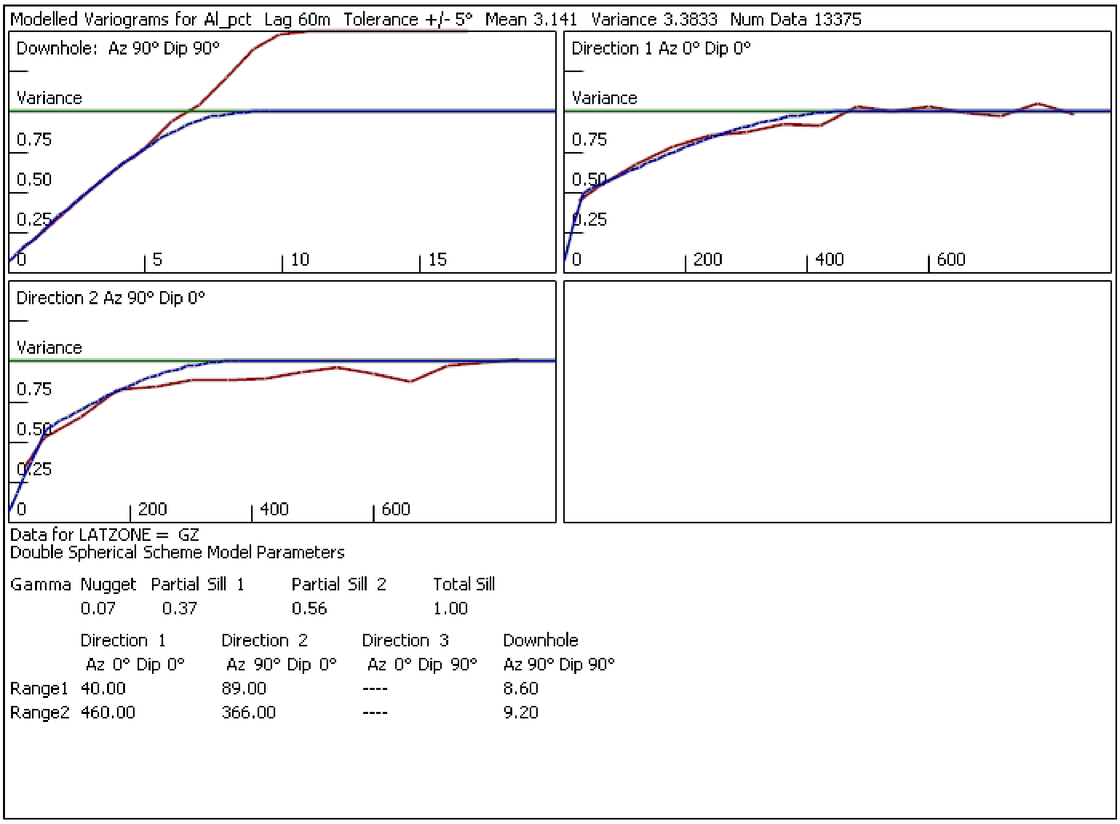
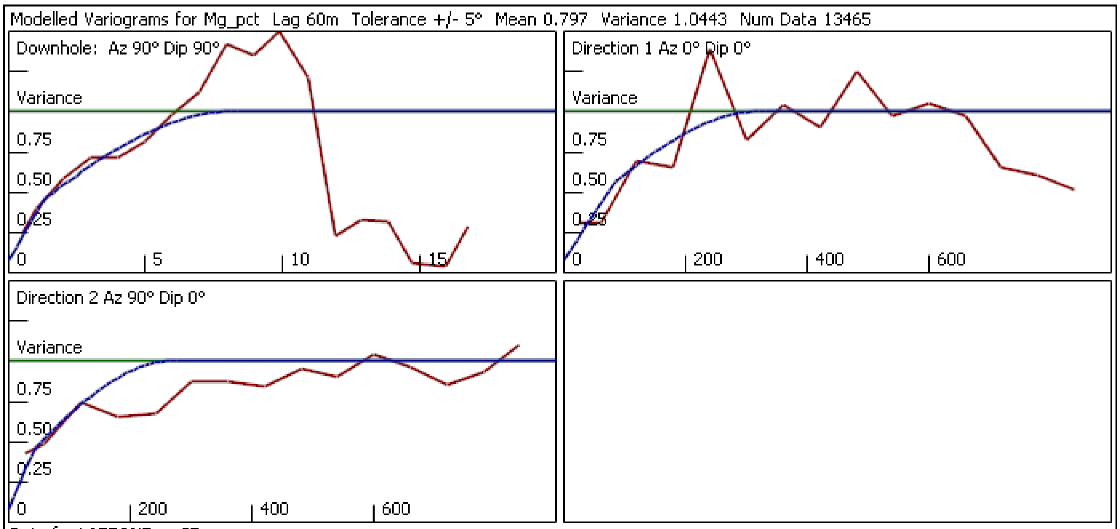



**Figure 14-58: GZ Al**

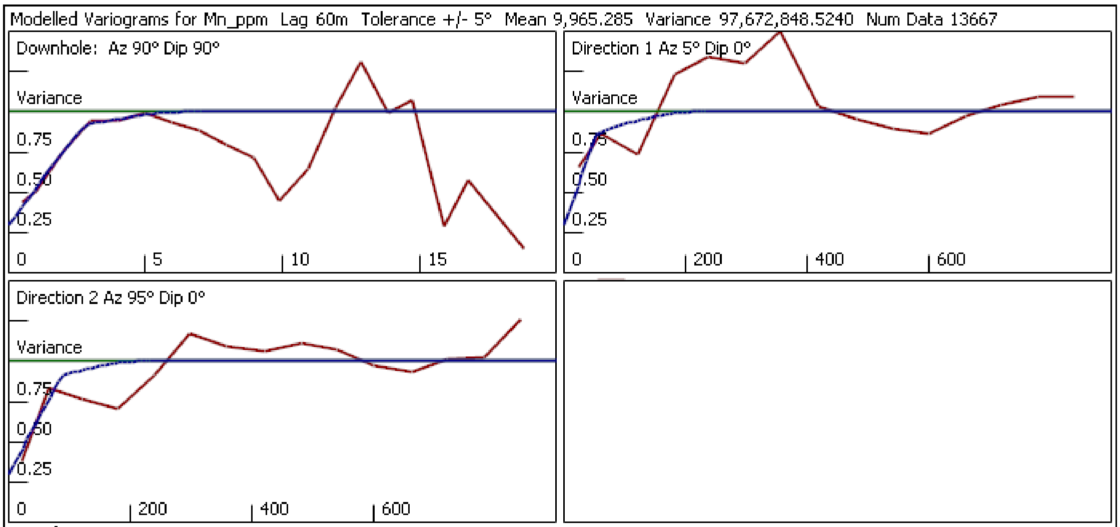



Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

| Gamma | Nugget | Partial Sill 1 | Partial Sill 2 | Total Sill |
|---|---|---|---|---|
| | 0.08 | 0.21 | 0.71 | 1.00 |

| | Direction 1 Az 0° Dip 0° | Direction 2 Az 90° Dip 0° | Direction 3 Az 0° Dip 90° | Downhole Az 90° Dip 90° |
|---|---|---|---|---|
| Range1 | 128.57 | 59.00 | ---- | 2.00 |
| Range2 | 320.00 | 267.00 | ---- | 8.10 |

**Figure 14-59: GZ Mg**




Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

| Gamma | Nugget | Partial Sill 1 | Partial Sill 2 | Total Sill |
|---|---|---|---|---|
| | 0.3 | 0.50 | 0.20 | 1.00 |

| | Direction 1 Az 5° Dip 0° | Direction 2 Az 95° Dip 0° | Direction 3 Az 5° Dip 90° | Downhole Az 90° Dip 90° |
|---|---|---|---|---|
| Range1 | 80.00 | 128.57 | ---- | 4.00 |
| Range2 | 225.00 | 225.00 | ---- | 6.80 |

**Figure 14-60: GZ Mn**

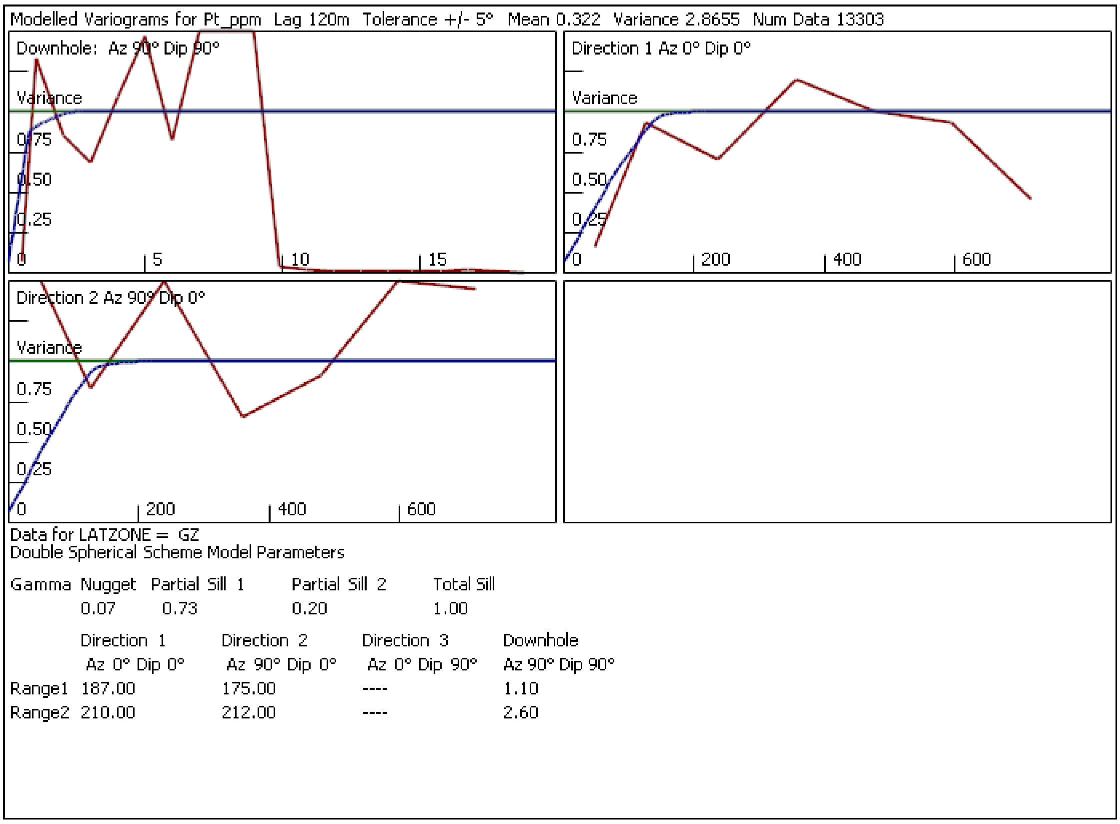



**Figure 14-61:GZ Pt**

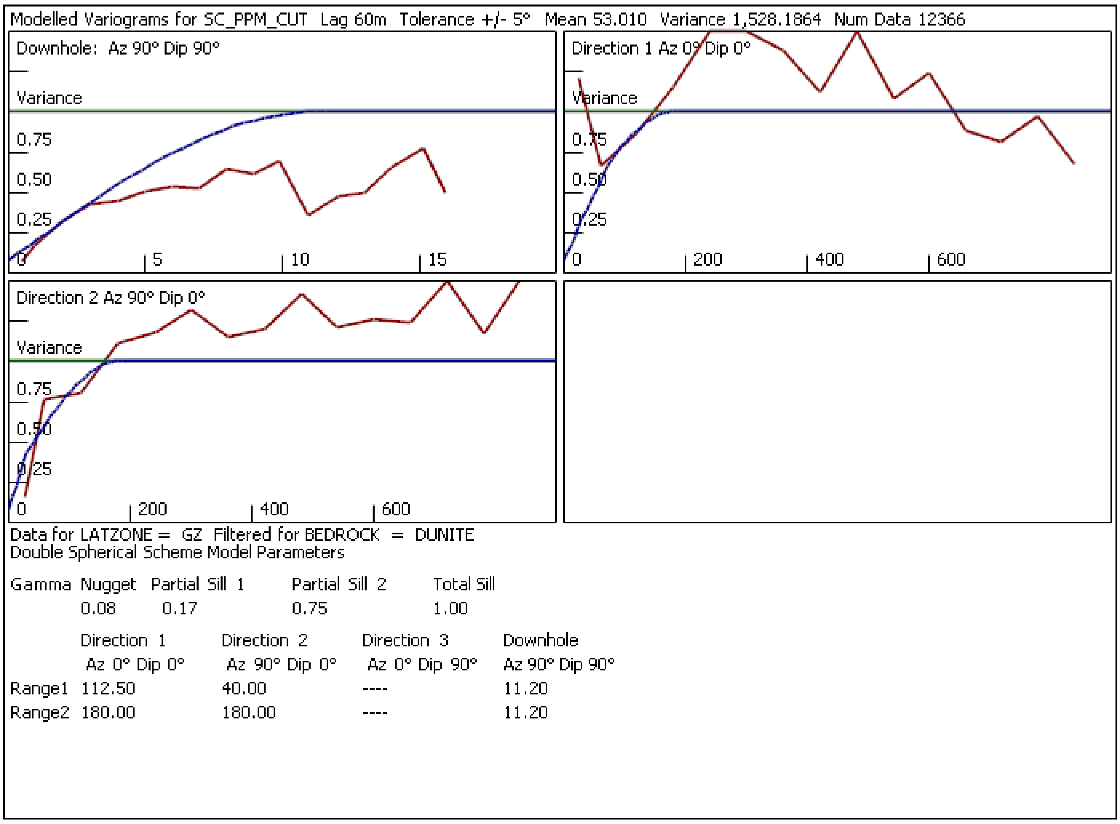



**Figure 14-62: GZ Sc**

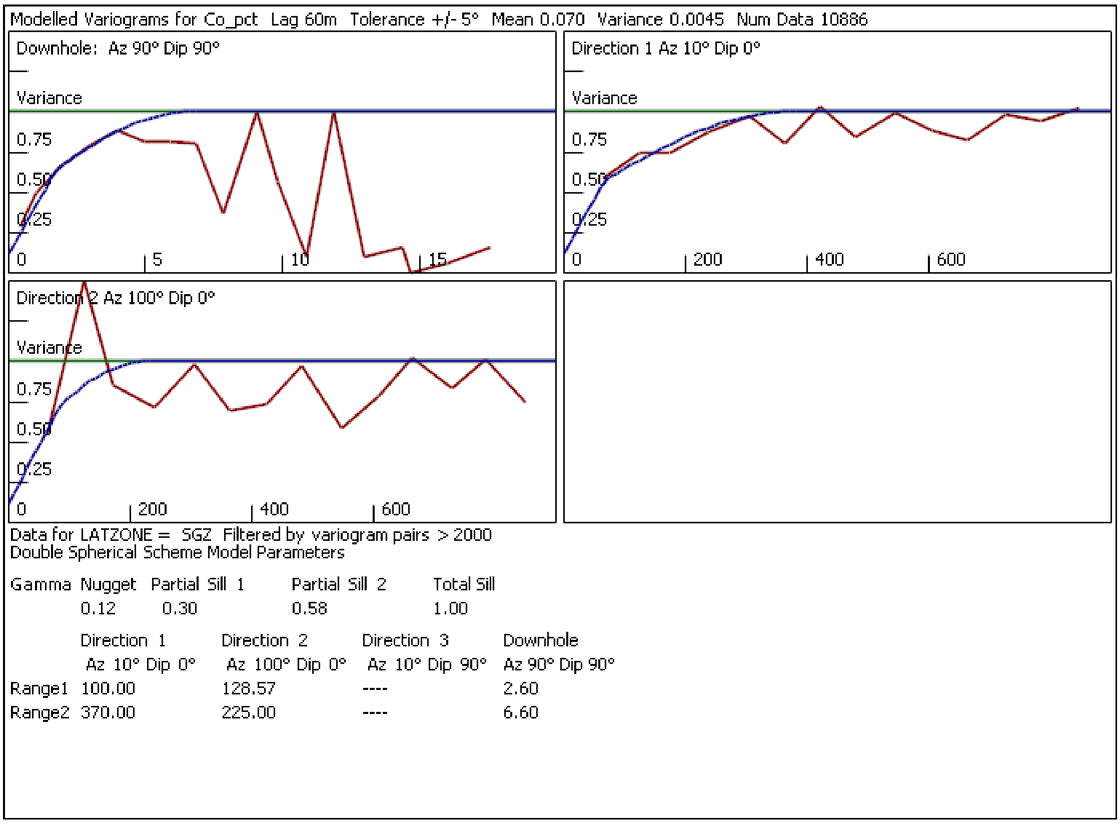



**Figure 14-63: SGZ Co**

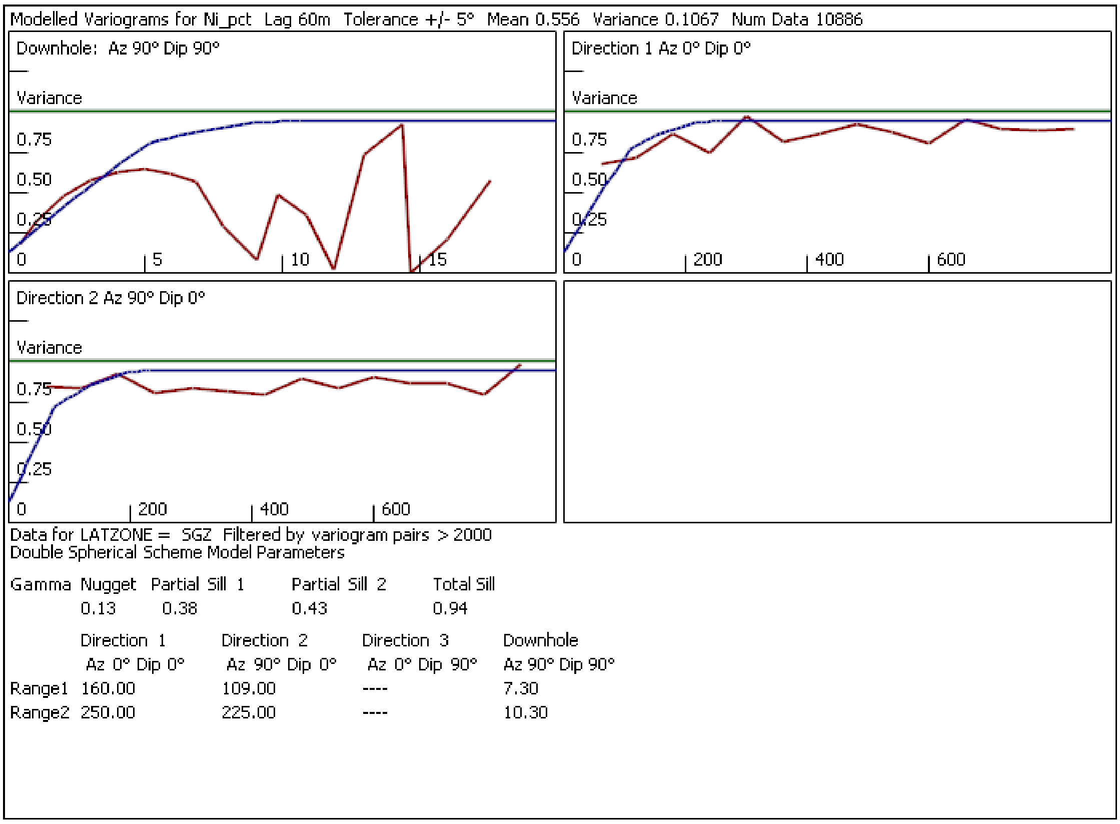



**Figure 14-64: SGZ Ni**

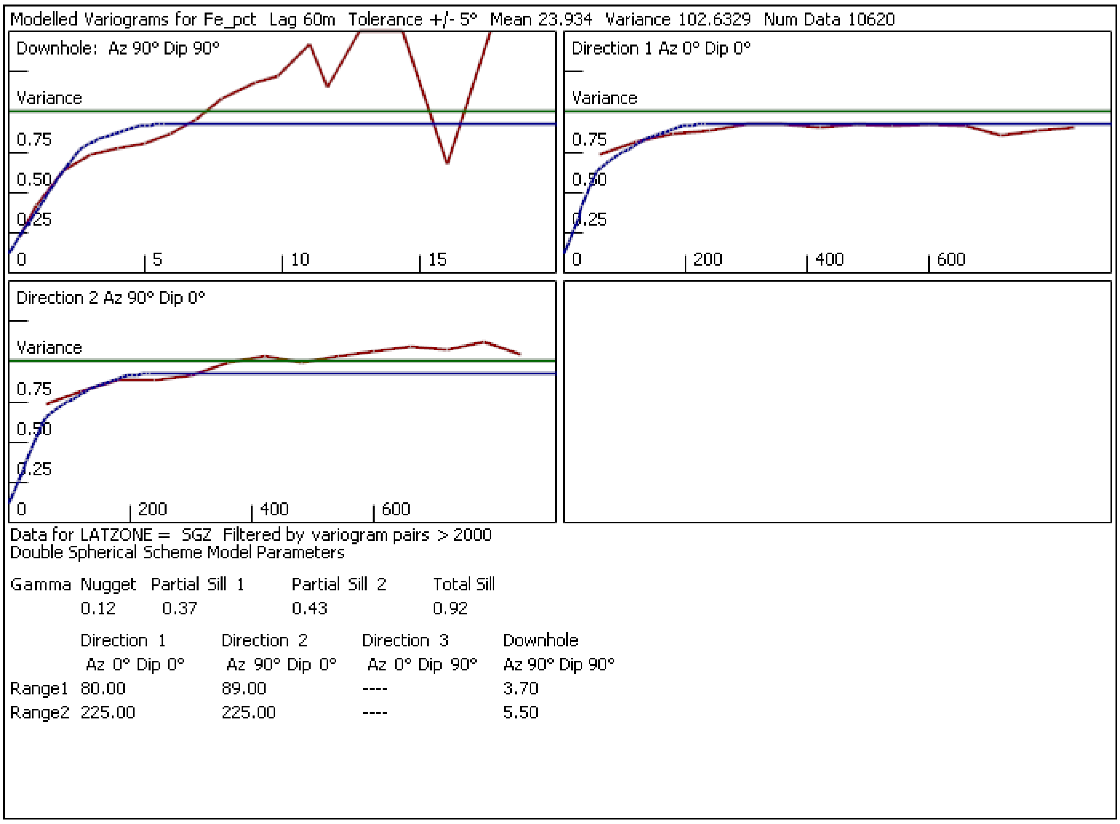



**Figure 14-65: SGZ Fe**

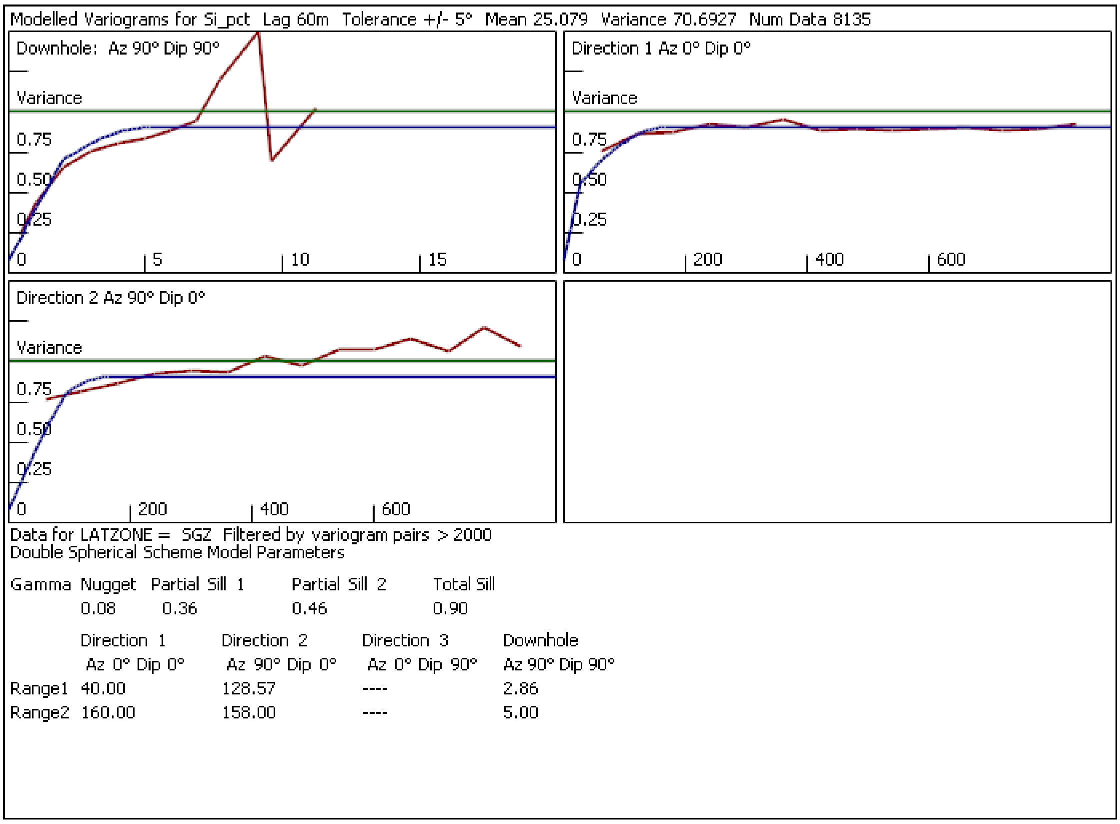



**Figure 14-66: SGZ Si**

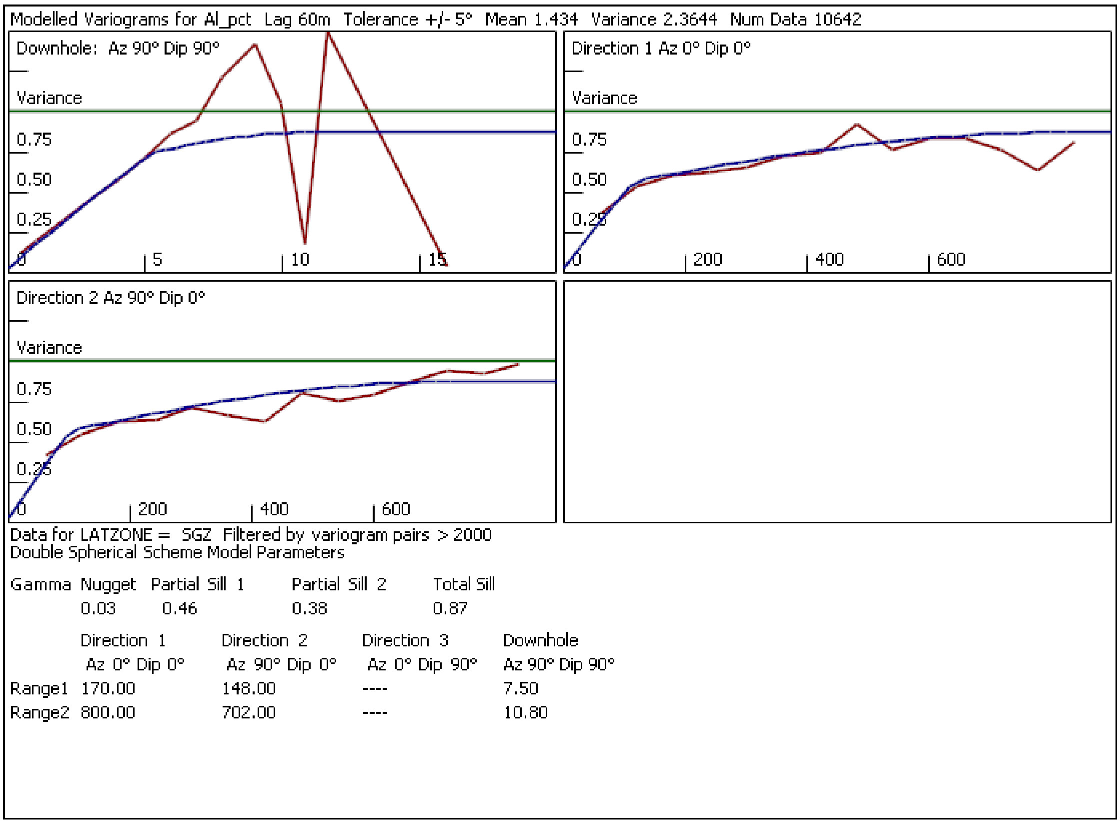



**Figure 14-67: SGZ Al**

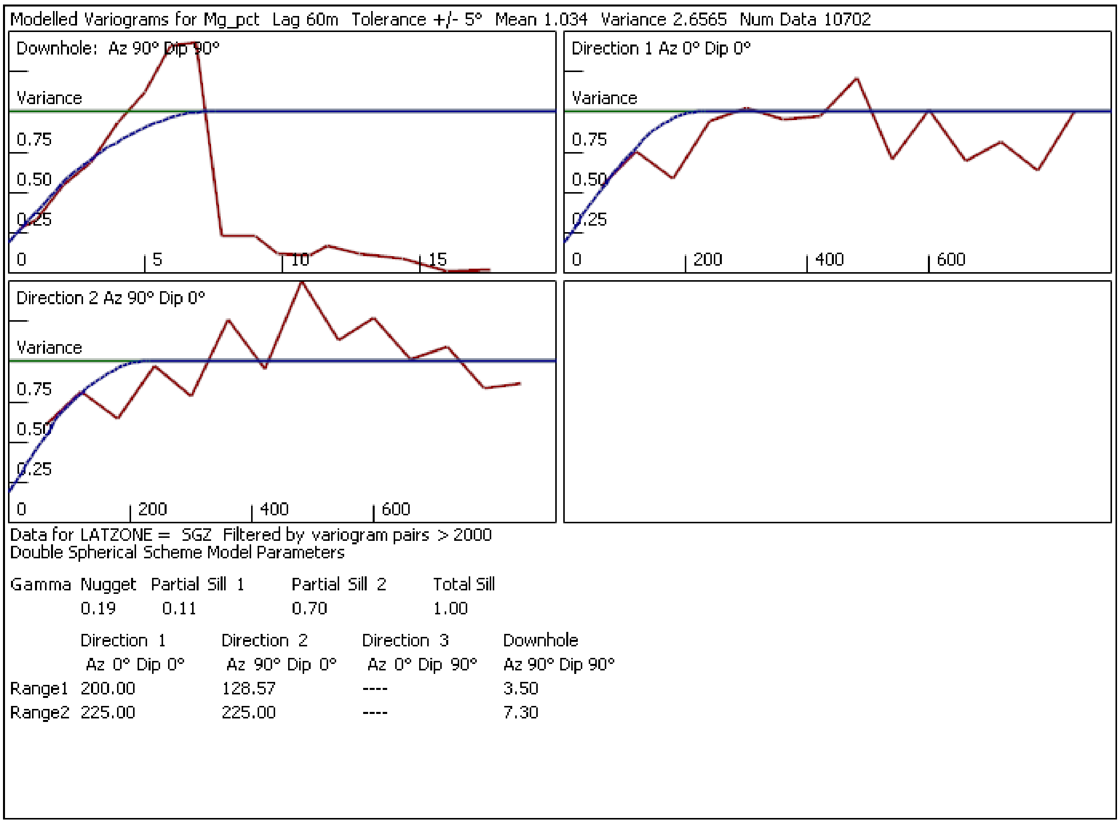



**Figure 14-68: SGZ Mg**

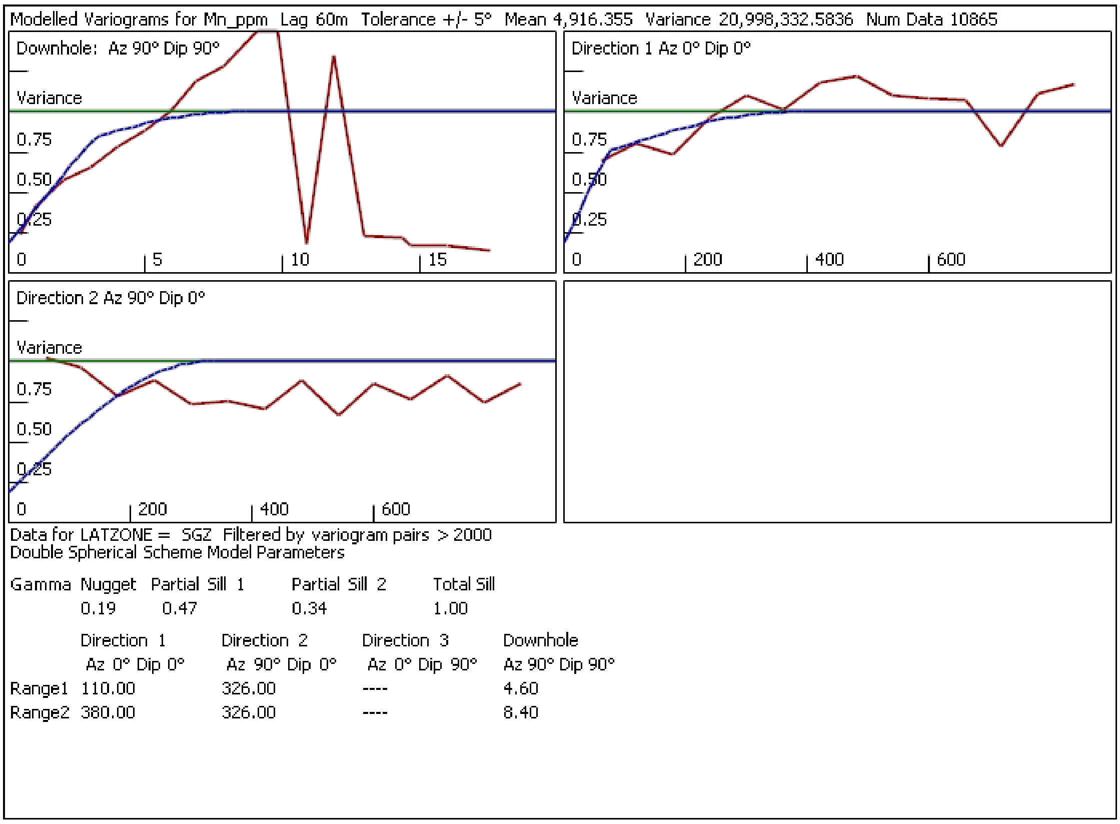



**Figure 14-69: SGZ Mn**

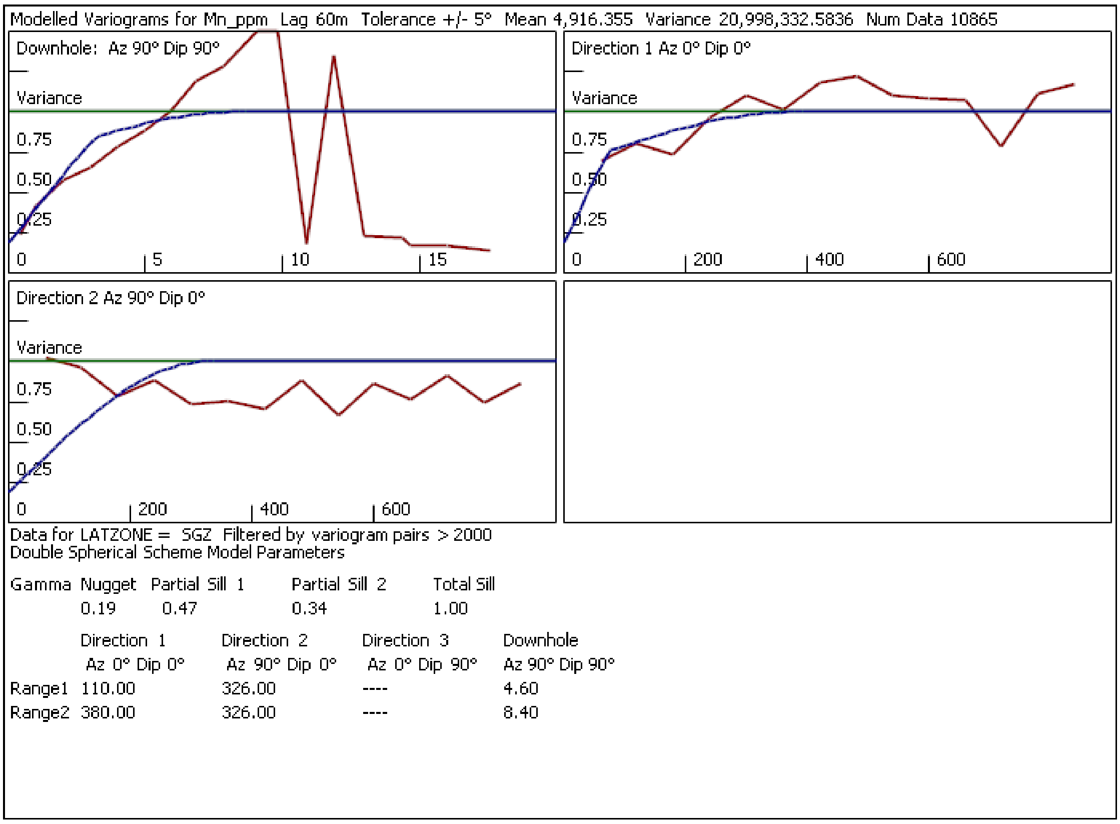



**Figure 14-70: SGZ Pt**

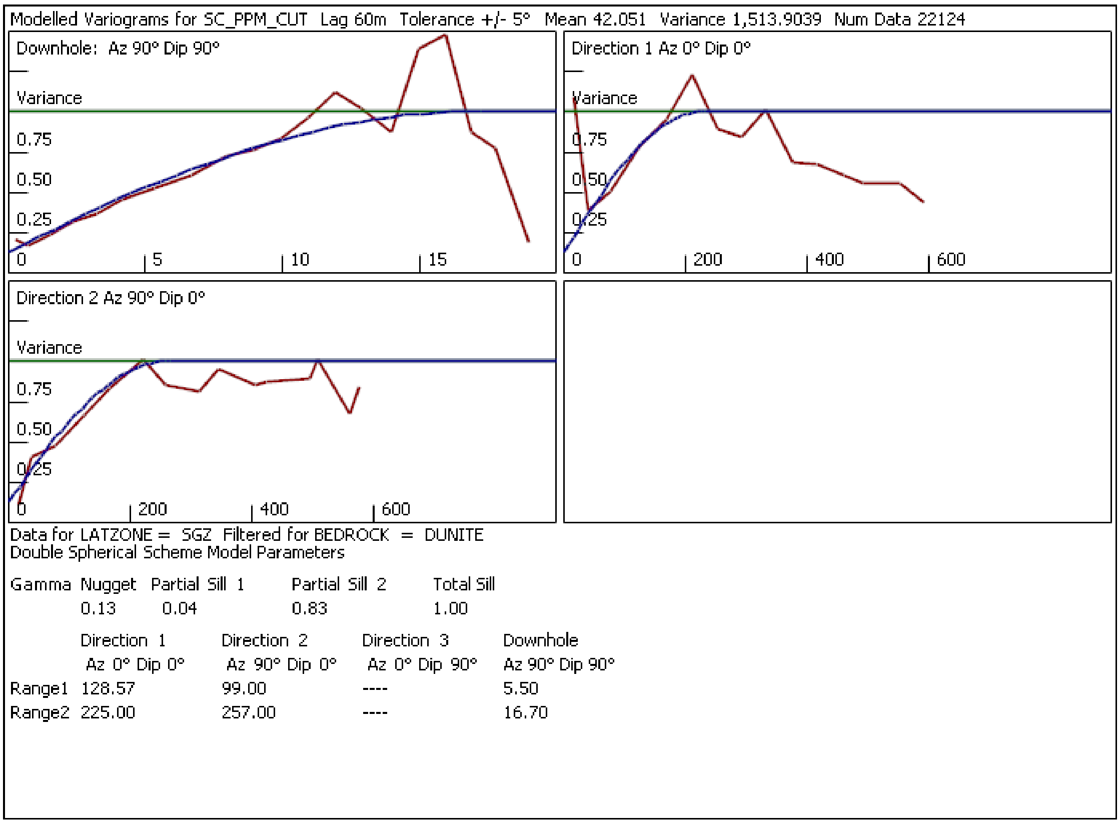



**Figure 14-71: SGZ Sc**

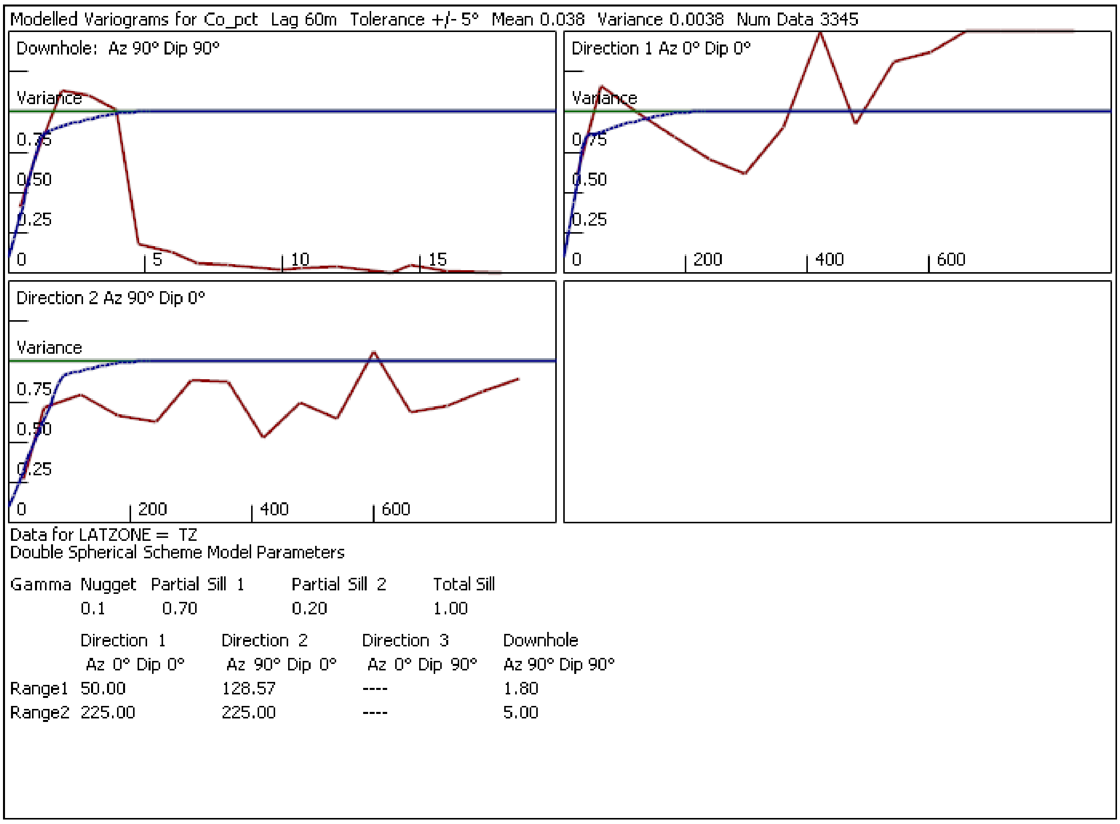



**Figure 14-72: TZ Co**

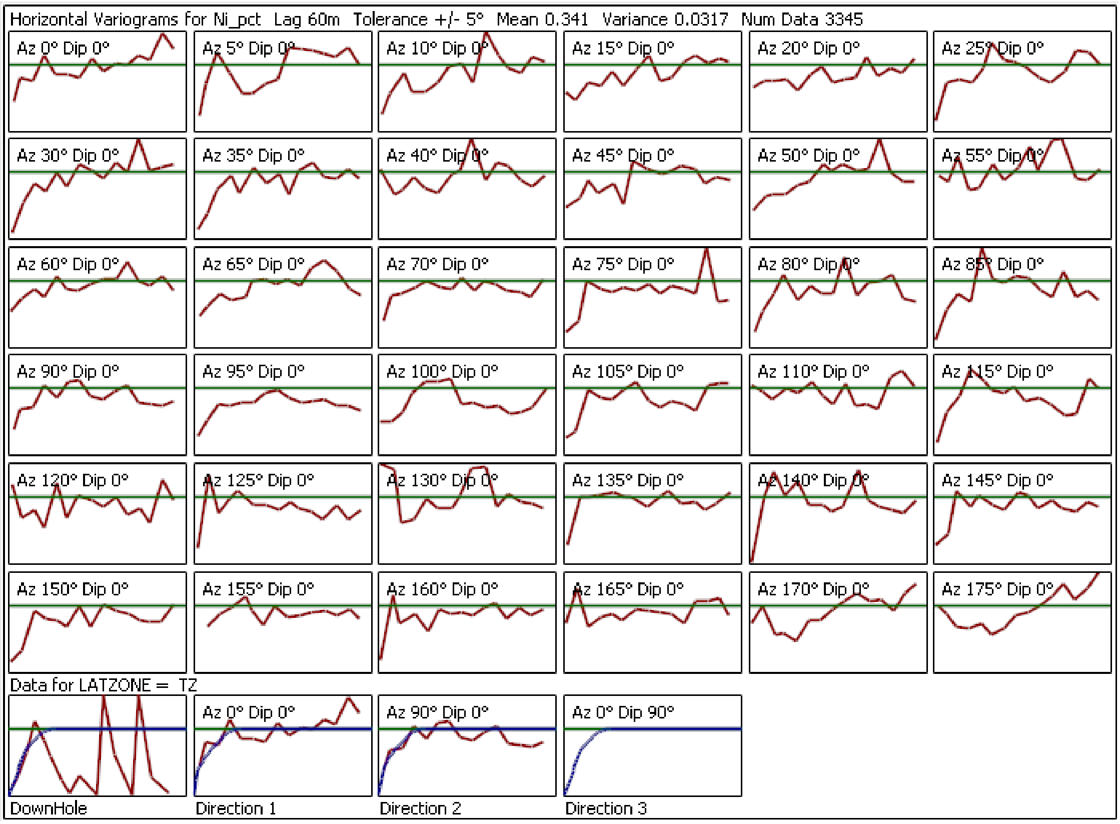



**Figure 14-73: TZ Ni**

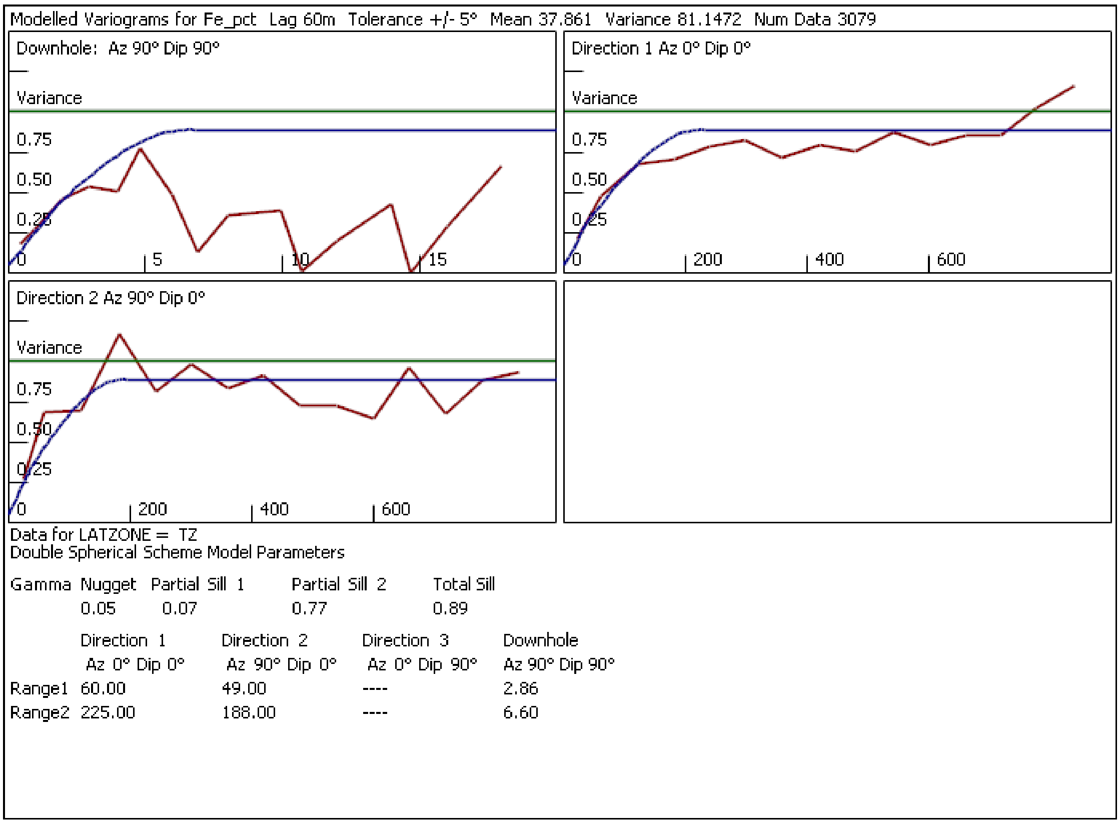



**Figure 14-74: TZ Fe**

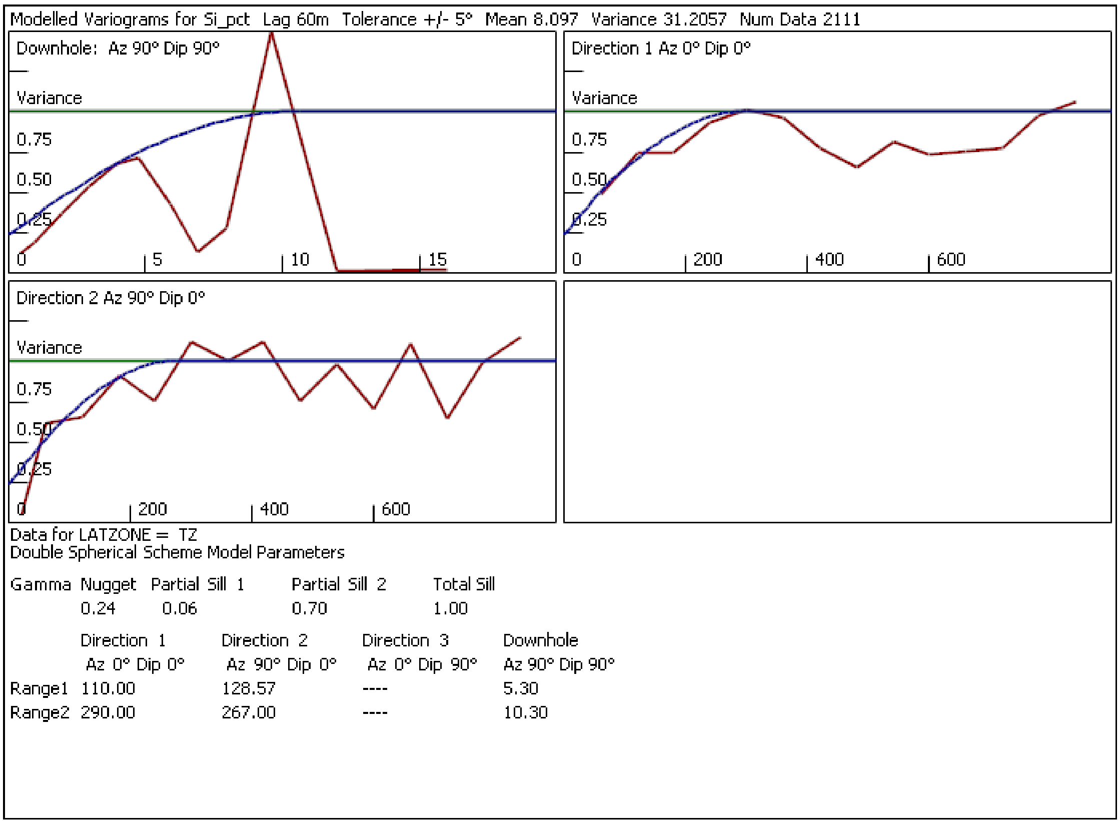



**Figure 14-75: TZ Si**

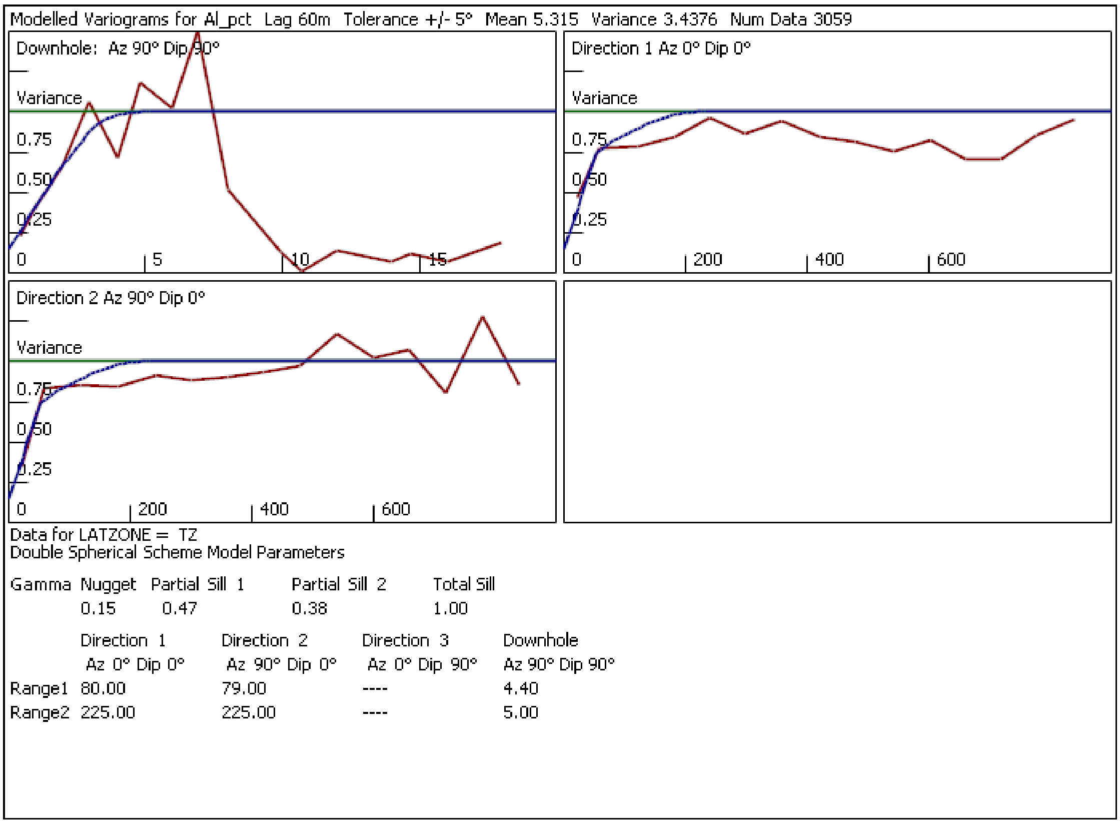



**Figure 14-76: TZ Al**

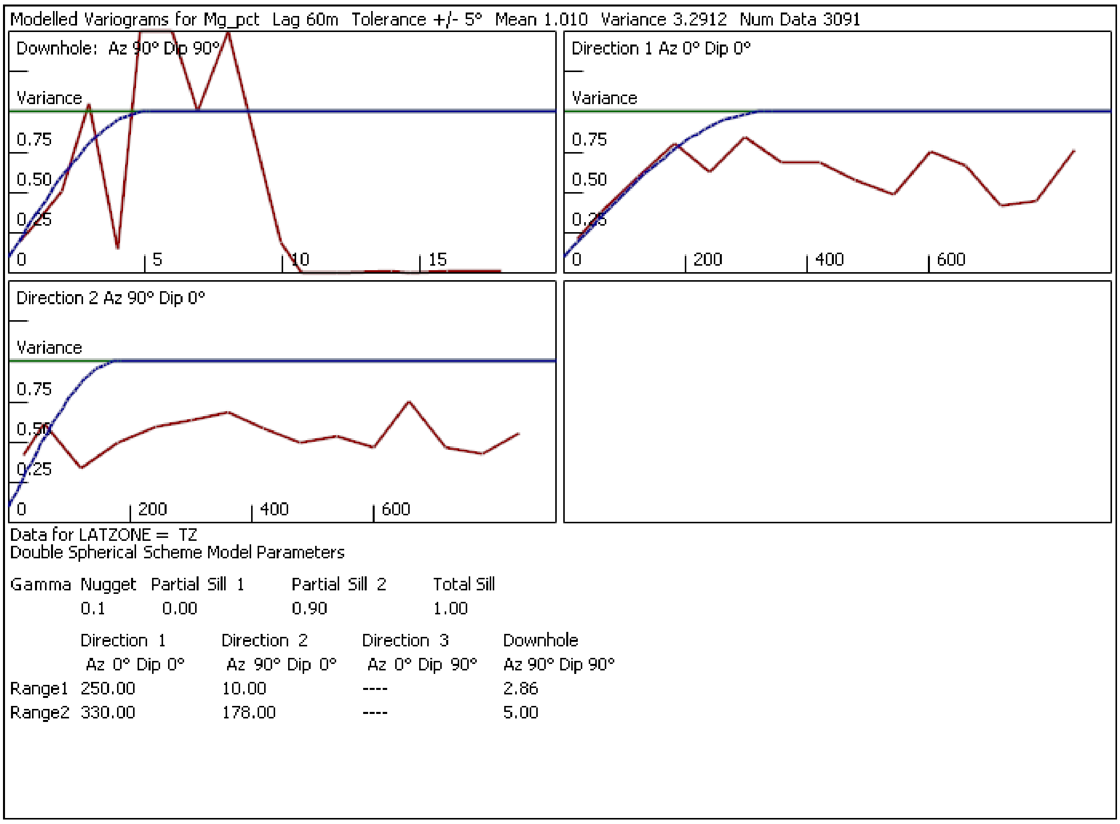



**Figure 14-77: TZ Mg**

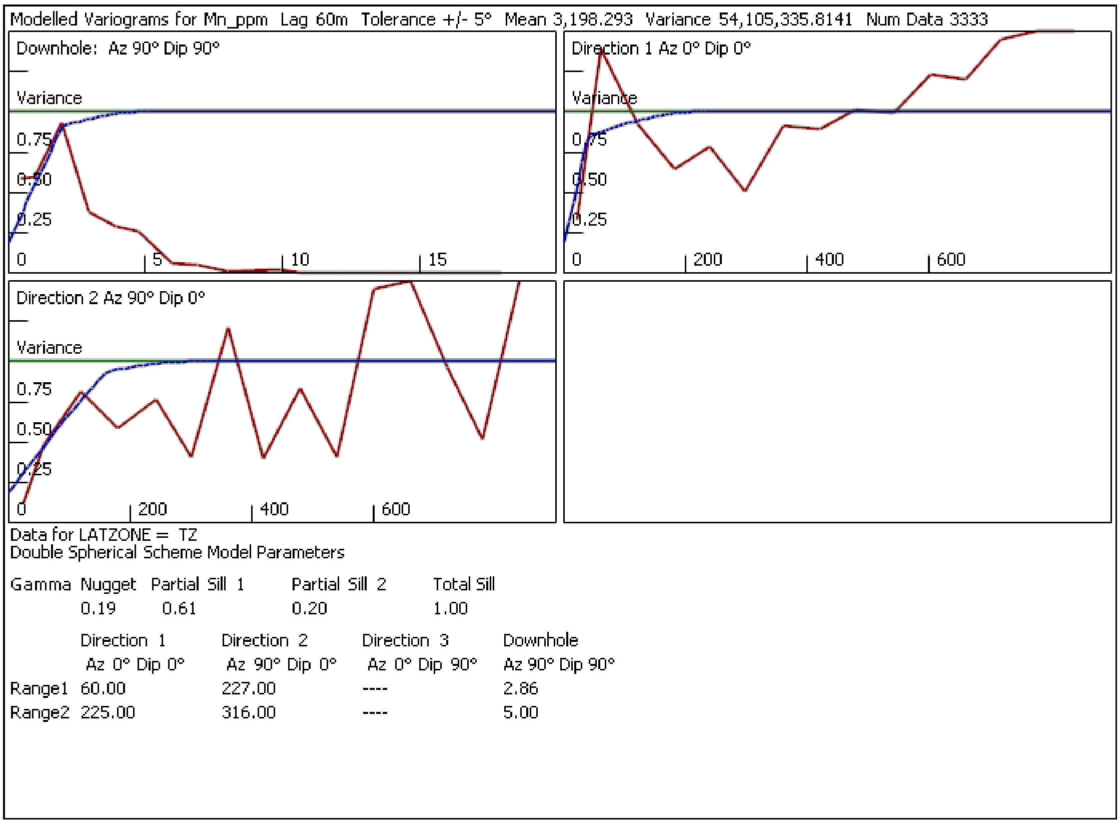



**Figure 14-78: TZ Mn**

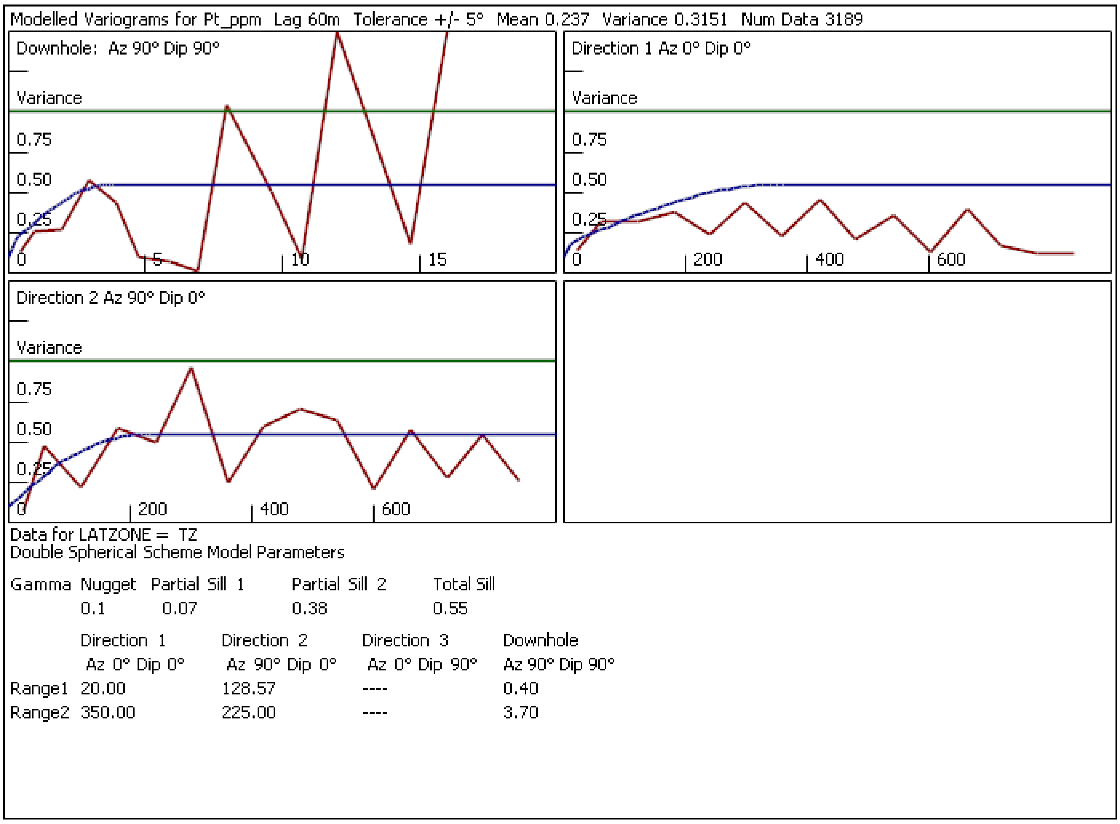



**Figure 14-79: TZ Pt**

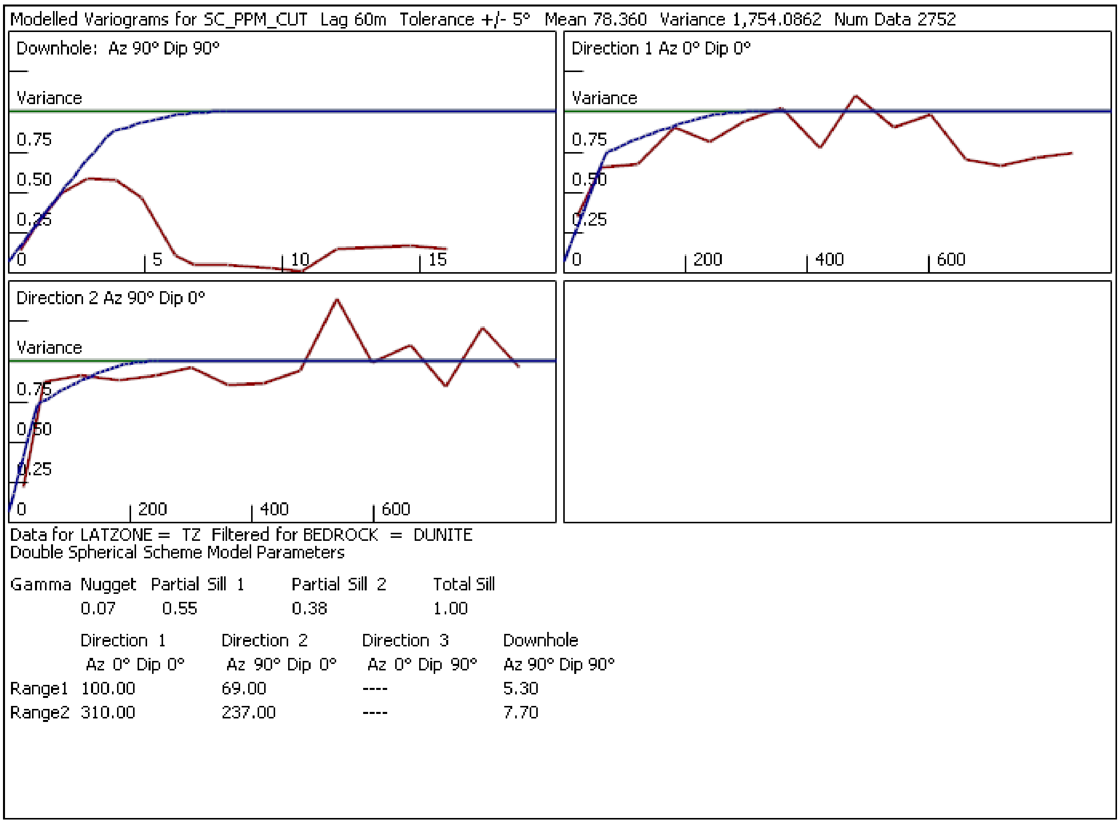



**Figure 14-80: TZ Sc**

### 14.2.9 Resource estimation procedures

All Resource Modelling has been carried out using Micromine 2016.1 Software.

Initially a three-dimensional rock model was generated with sub-division based on the LATZONE domains, the outline of the Dunite Complex and constrained by a Limit of Mineralisation envelope.

Subsequently a high grade Cobalt Domain was defined within the GZ zone, and a high Scandium domain was also defined.

Block grades estimated for Cobalt and Nickel plus Sc, Pt, Fe, Si, Mg, Al and Mn were generated using Ordinary Kriging. Cu, Zn, Cr, As, Au, Pd and Ca were estimated using an anisotropic inverse distance squared interpolation method.

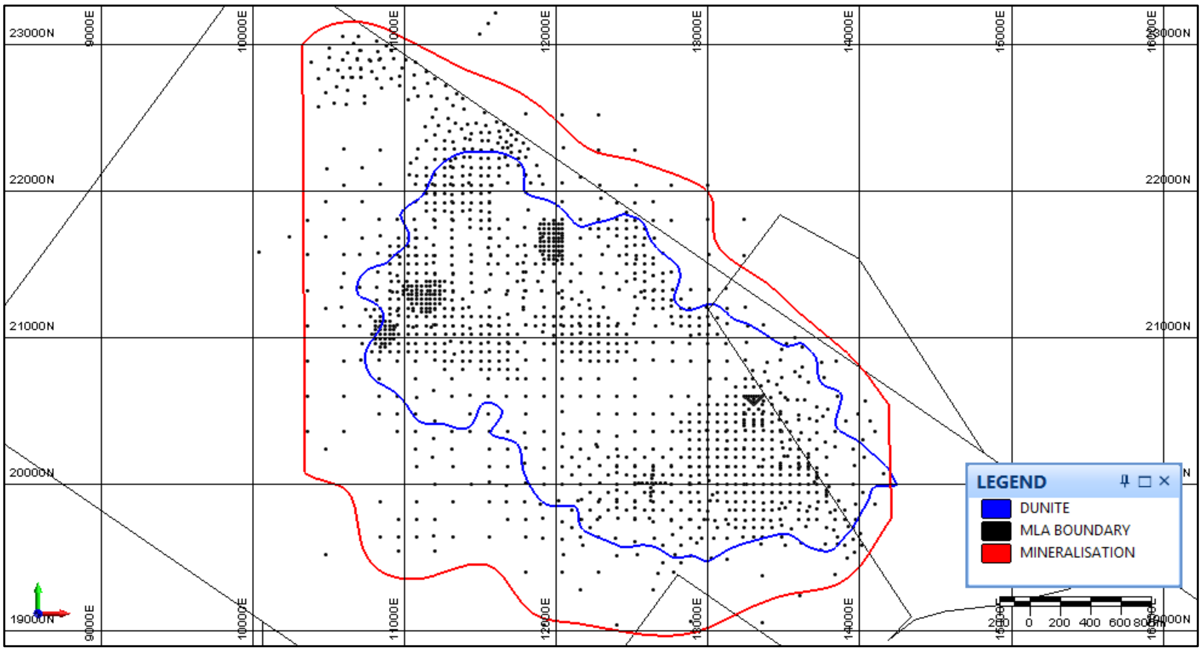



**Figure 14-81: Drill hole locations, Dunite limit and mineralisation limits**

### 14.2.10 Drill hole spacing and block size definition

The variable drill hole spacing (see plan above) has been reviewed in a number of ways to assist in both resource classification (together with other output from kriging, such as kriging variance, samples used etc), and in deciding upon the block sizes to be used in the model.

By convention block size is selected as the best combination of X, Y, Z sizes that suit final SMU and mining bench selection plus conform to geostatistical expectations in terms of minimum block size in relation to drill hole spacing. Typically, for an Ordinary Kriging estimation methodology, one quarter to one half of the spacing would be suitable.

In the case of Syerston, spacing varies from small areas of very close spacing (<15m) to wide spaced (>200m) areas. There are clear areas with approximate grids of 60x60 and 120x120, but there are also more erratically drilled areas that are hard to quantify.

With modern mining software (Micromine 2016.1 is used for estimation in this is case), there is no particular reason to have a single block size for a deposit, which will be either too large or too small for some areas. Rather, a hybrid block size model can be generated with the parent cell size adjusted to suit the local drill spacing. In terms of the mining software model prototypes this is equivalent to making the largest block size the overall parent size, while all other combinations of block sizes will be considered as subcells. There is also sub-celling to follow topographic and domain boundaries, so this is the normal case in rock model setup.

For the Syerston case three different block sizes are proposed, depending on average drill spacing. The areas for each size need to reasonably large to avoid over-complicating the setup of the model. Following various analyses which quantify the spacing, the small block size would be used for the very close spaced drilling areas, a medium block size for the typical 60x60 drill pattern, and a larger block size for 120x120 and greater areas.

An additional reason for minimising block size came out of a comparison of previous kriged models with a Conditional Simulation (CS) model, which suggested that the kriged models were over-smoothed compared to an expected recoverable SMU model.

Consequently, block sizes were chosen to be as small as feasible and geostatistically acceptable. Initially blocks of 5x5, 10x10 and 20x20 (the previous model block size) were trialled. However, this results in blocks which are one sixth of the spacing for most cases (10m for 60x60 and 20m for 120x120). Whilst blocks of this size may make the block grade distribution mover closer to the CS and data distributions (depending as well on other estimation parameters), these blocks are rather small to be justified from a geostatistical point of view. A second run using 15x15 and 30x30 has been used (representing one quarter of drill spacing in both cases); with appropriate search and kriging parameters, there is minimal difference between the grade distribution of these and the 10x10 and 20x20 versions.

The block height of 2m was based on advice about a likely open pit working bench height.

The graphs below show the results of a 2-dimensional modelling exercise using collar locations and reviewing numbers of points found and kriging variance; a pseudo-variogram model was used, purely to get relative kriging variances, which depend solely on number and orientation of samples.

Following this, areas were digitised to control the size of blocks in the rock model. The grids are shown on the following pages.

Table 14-17 shows the relative proportions of each block size; interestingly the main mineralised zones (FZ, and to a lesser extent TZ and SGZ) are better drilled and have higher representation of the smaller block sizes.

**Table 14-17: Proportion of blocks in various block sizes**

| LATZONE | 5x5 | 15x15 | 30x30 |
|---|---|---|---|
| AV | 1% | 36% | 63% |
| OVB | 1% | 26% | 73% |
| TZ | 5% | 56% | 39% |
| GZ | 6% | 61% | 33% |
| SGZ | 5% | 56% | 40% |
| SAP | 3% | 42% | 55% |

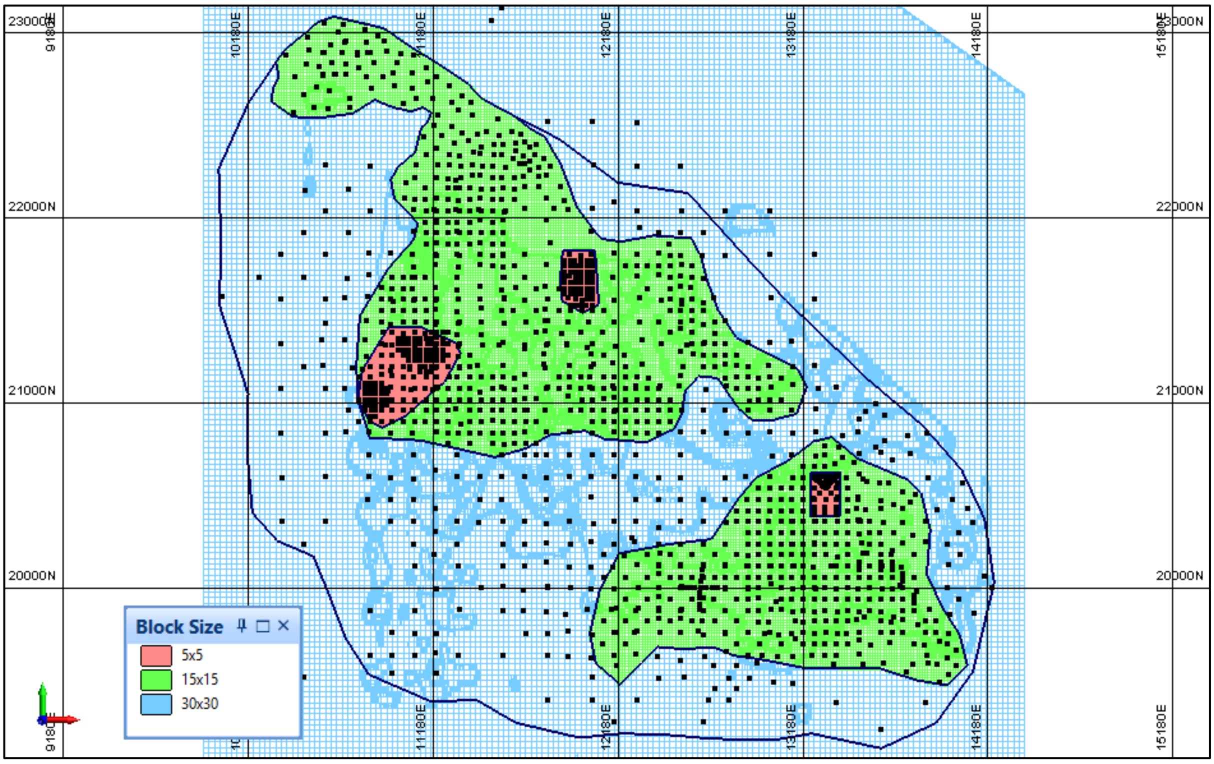



**Figure 14-82: Final block size distribution**

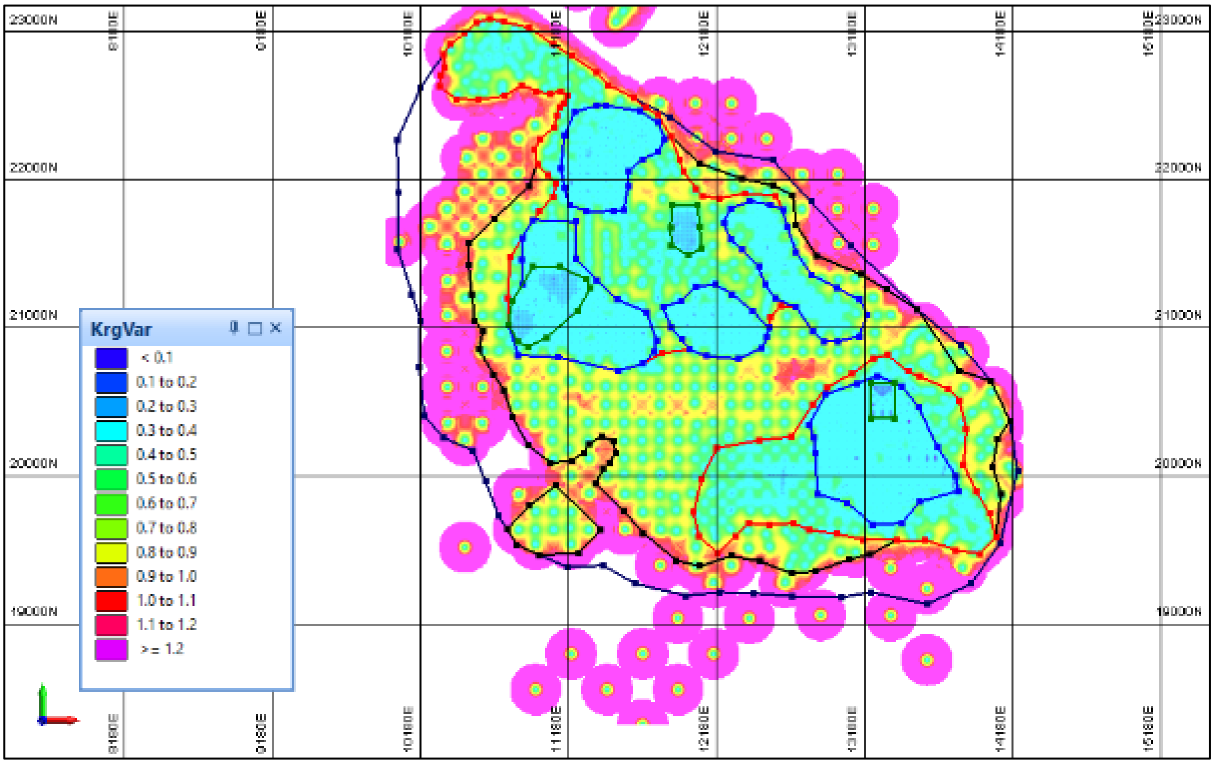



**Figure 14-83: 2D "kriging variance" of collar locations**

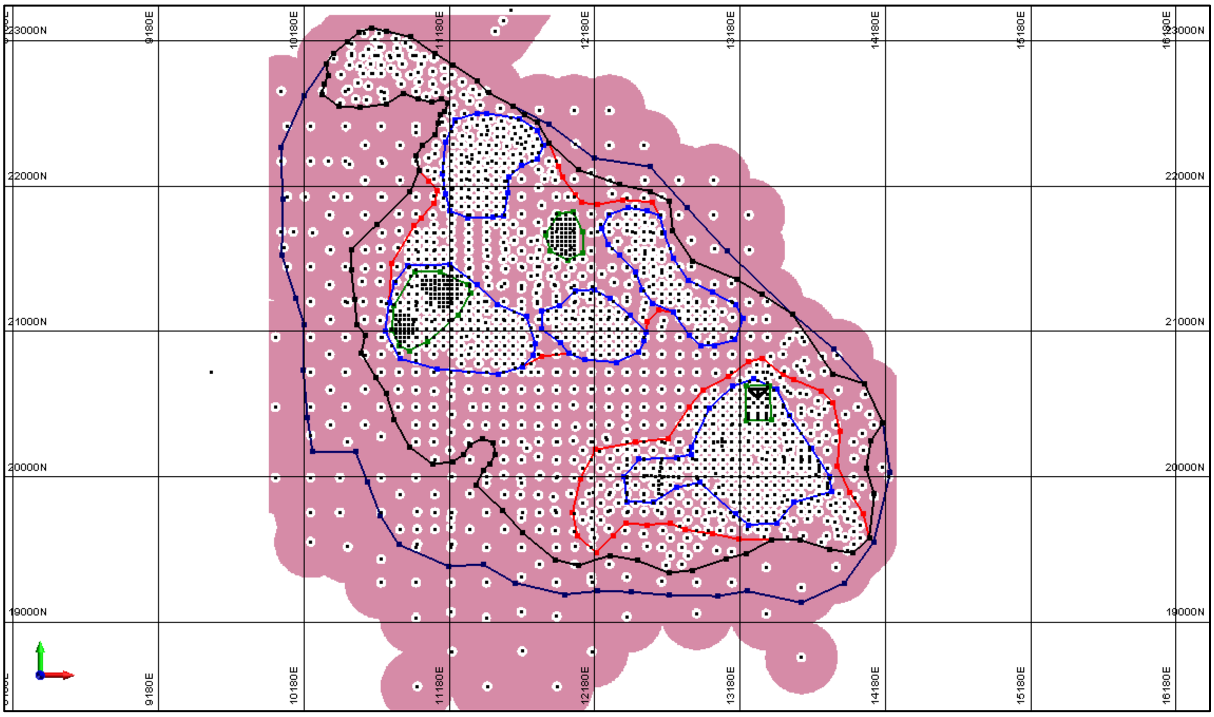



**Figure 14-84: Average distance <35 m (white areas)**

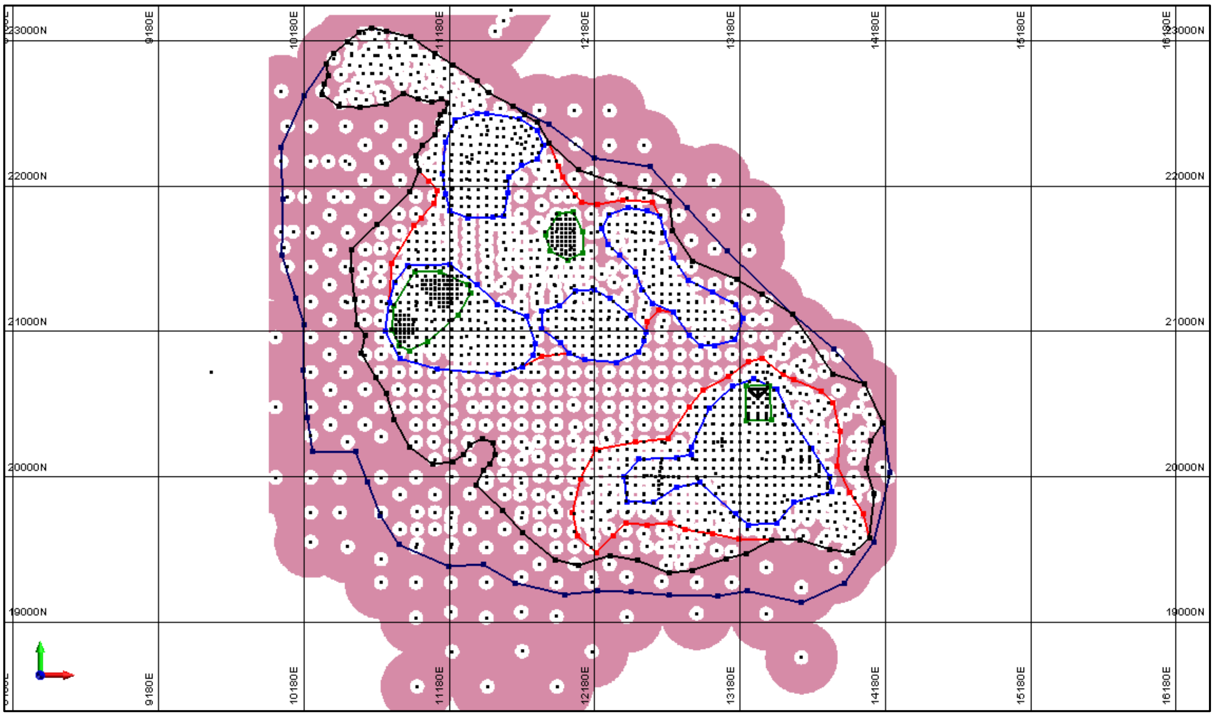



**Figure 14-85: Average distance <50 m (white areas)**

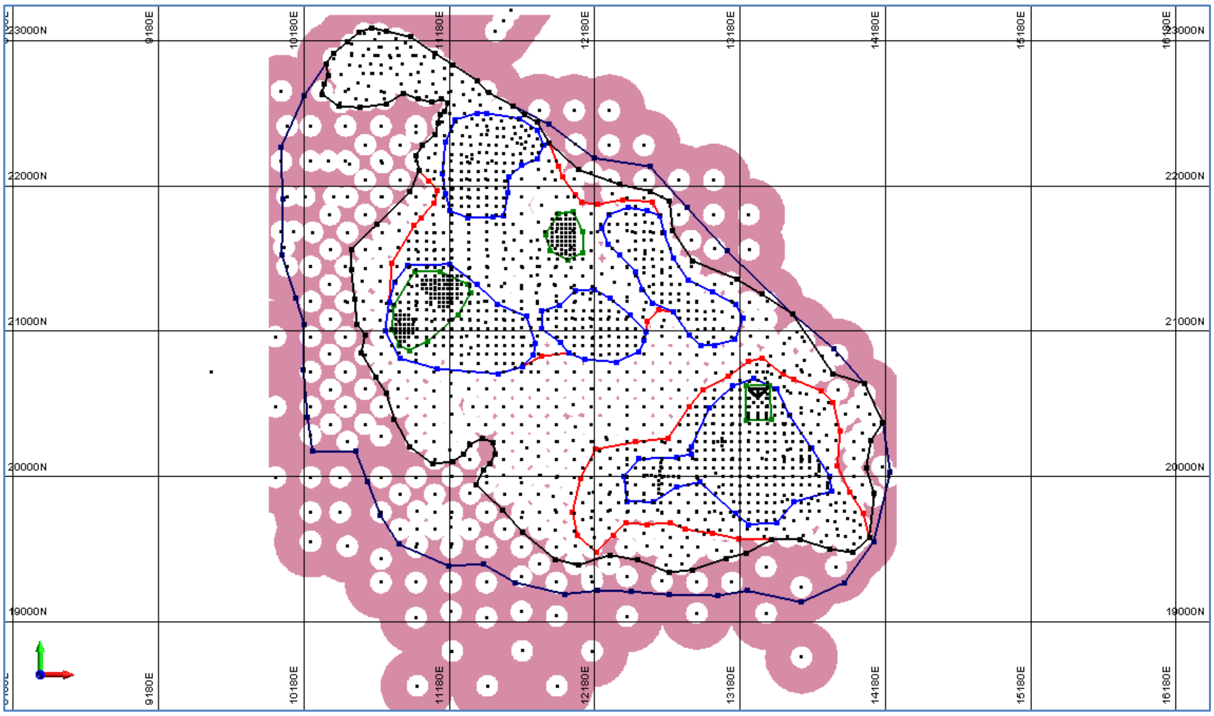


**Figure 14-86: Average distance <75 m (white areas)**

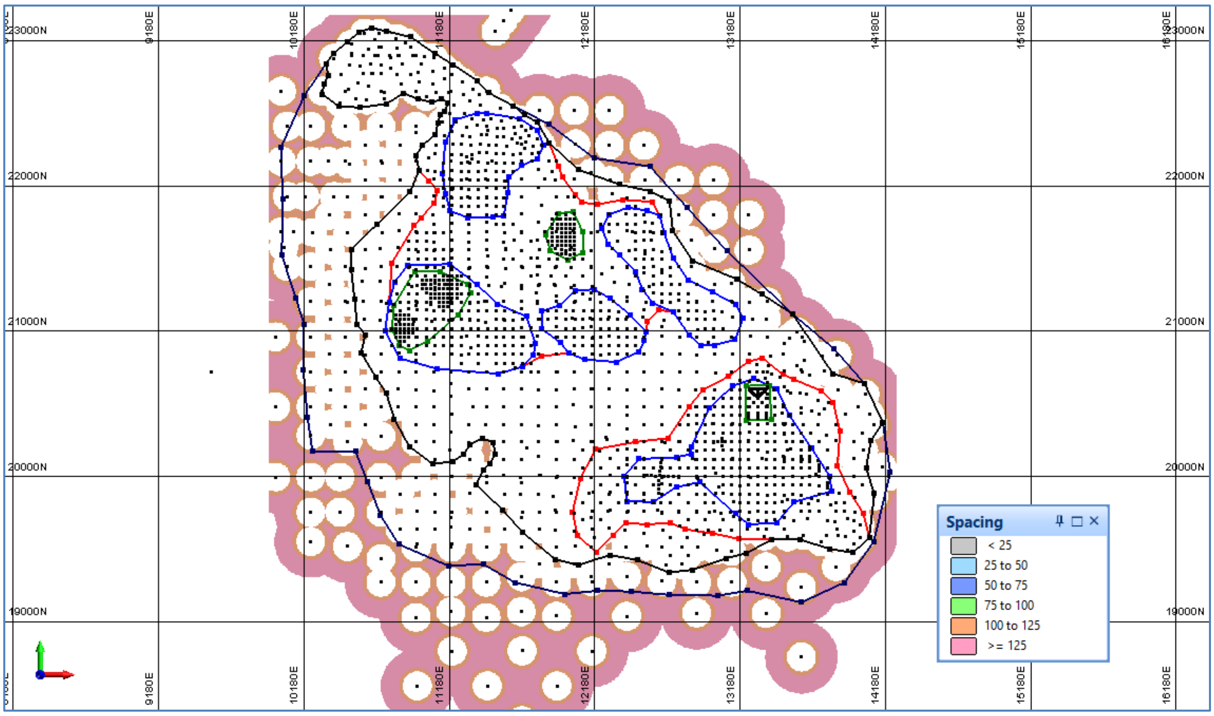



**Figure 14-87: Average distance <100 m (white areas)**

## 14.2.11 Rock model

Laterite zone boundaries have been re-interpreted as strings on north–south section lines on the basis of the latest drilling results, and using a refined chemical definition of each zone.

These strings were linked to form surface wireframes (DTMs) representing:

- Base of Alluvium (AV)
- Base of Overburden (OVB)

- Base of Transition Zone (TZ)
- Base of Goethite Zone (GZ)
- Base of Siliceous Goethite Zone (SGZ).

The wireframe surfaces have then been smoothed by the Implicit Surface Modelling process within Micromine 2016.1. Software, which generates smoother surfaces while still honouring snapped drill hole points. In addition, algorithms were applied to eliminate surface overlaps. Also, in order to prevent extrapolation below the base of drill holes in Saprolite, a Base of Saprolite (SAP) surface was generated parallel to the Base of SGZ surface and 8 metres below it.

As a final step, wireframes were clipped to a limiting boundary.

Wireframes were filled above or below with blocks and combined in a controlled sequence to generate a 3D rock model.

**Table 14-18: Block model origin and block size parameters**

| MICROMINE FORMAT USING CENTROIDS | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| EAST | | NORTH | | RL | | BLOCK SIZE | | | NUMBER OF BLOCKS | | | MINIMUM SUBCELL SIZE | | |
| MIN | MAX | MIN | MAX | MIN | MAX | EAST | NORTH | RL | EAST | NORTH | RL | EAST | NORTH | RL |
| 9955 | 14395 | 18315 | 23295 | 221 | 329 | 30 | 30 | 2 | 149 | 167 | 55 | 0.5 | 0.5 | 0.2 |
| | | | | | | | | | | | | | | |
| DATAMINE FORMAT USING EDGES | | | | | | | | | | | | | | |
| EAST | | NORTH | | RL | | BLOCK SIZE | | | NUMBER OF BLOCKS | | | MINIMUM SUBCELL SIZE | | |
| MIN | MAX | MIN | MAX | MIN | MAX | EAST | NORTH | RL | EAST | NORTH | RL | EAST | NORTH | RL |
| 9940 | 14410 | 18300 | 23310 | 220 | 330 | 30 | 30 | 2 | 149 | 167 | 55 | 0.5 | 0.5 | 0.2 |

Blocks intersected by the topographic surface were sub-celled to 0.2 m in the vertical direction

Typical cross sections and plans through the rock model with the wireframe surfaces and drill hole coded LATZONEs are shown below.

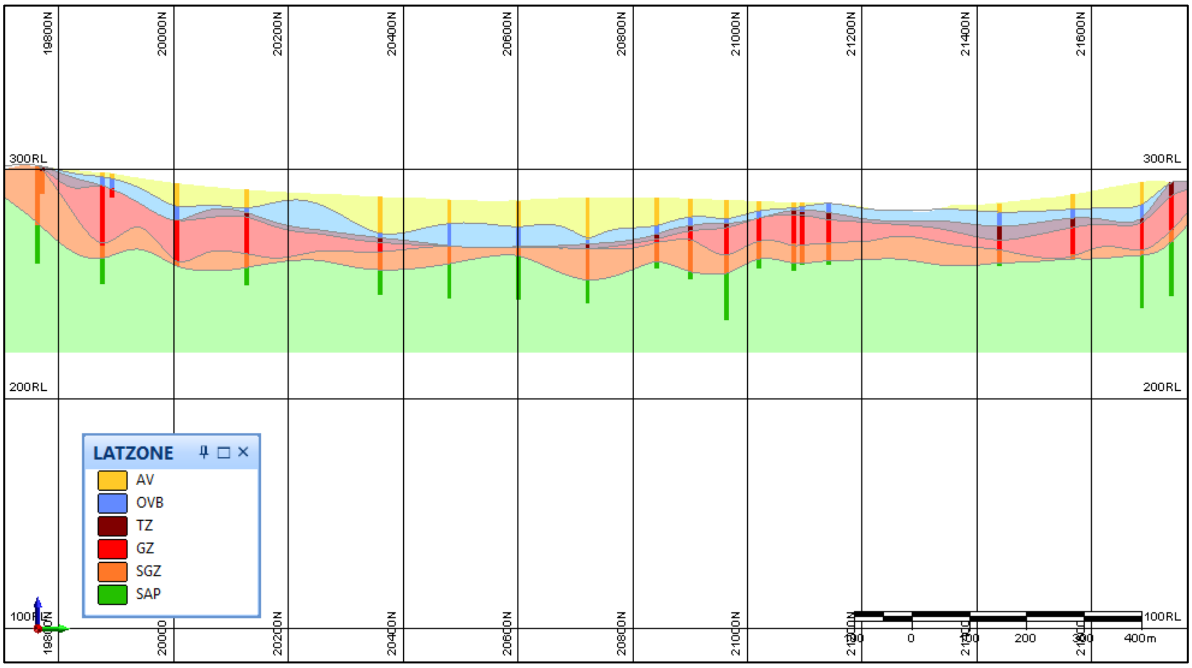



**Figure 14-88: Typical Easting section**

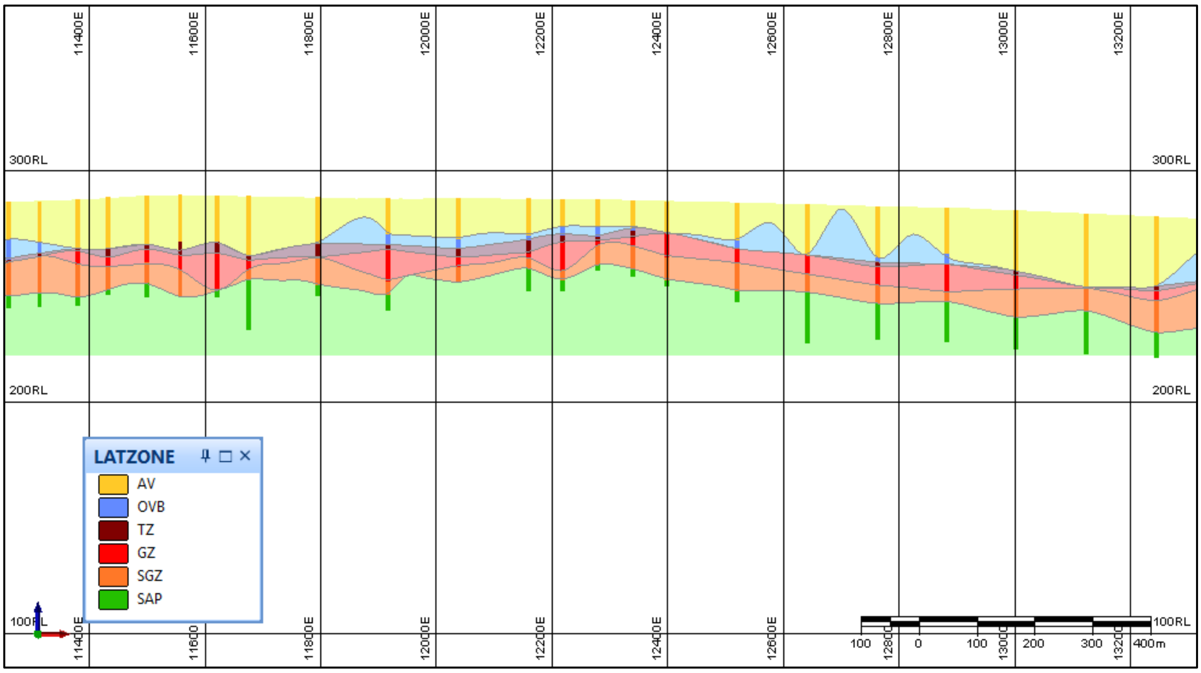



**Figure 14-89: Typical Northing section**

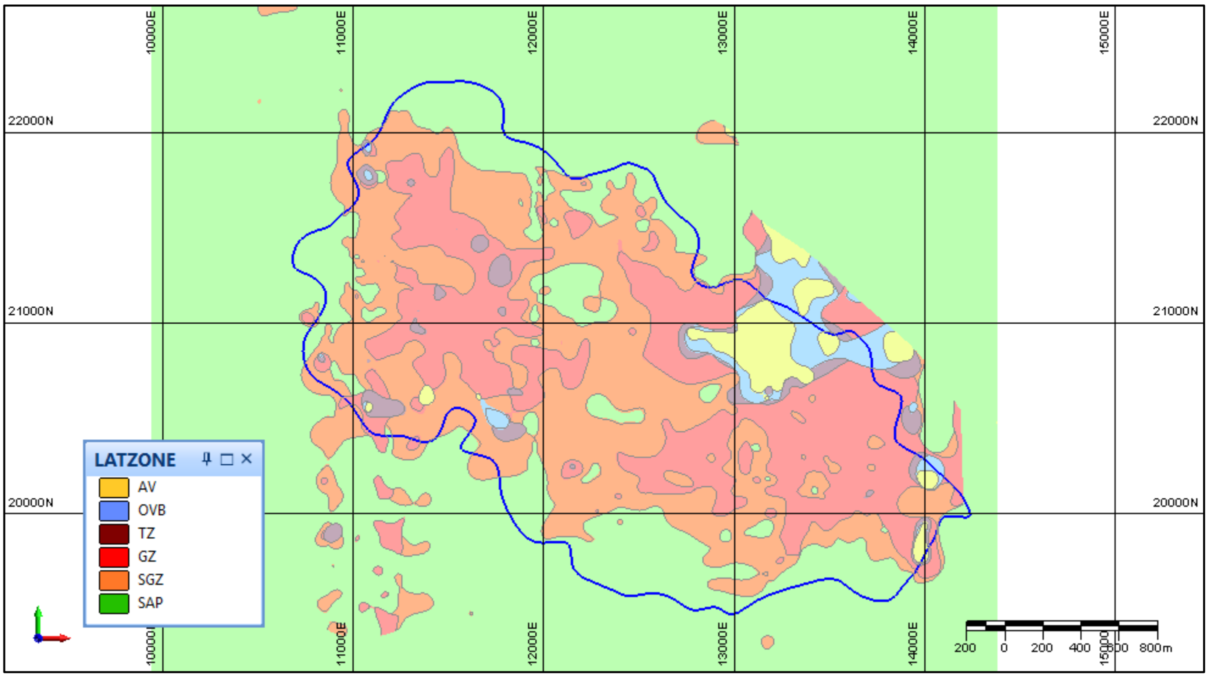



**Figure 14-90: Typical level plan**

### 14.2.12 Interpolation parameters

Search parameters are based on the variogram ranges, with the first search ellipse pass approximating the 60x60 drill spacing (and also within the first range of the double spherical scheme variogram models) and the second pass designed to populate blocks from data on the 120 x 120 pattern. This search is also within the overall range of the variograms. Search ellipses are generally isotropic and use a “flattening” or “unfolding” process to simplify the specification of slight variations of dip needed to follow the undulations in the surfaces. This is illustrated in the cross section below.

**Figure 14-91: Typical section showing unfolded (top) vs in-situ blocks**

Search parameters are the same for Ordinary kriging and Inverse Distance interpolation are shown in Table 14-19.

**Table 14-19: Search parameters**

| | Blocks | Search Ellipse | | | Samples | | Holes | Per Hole | |
|---|---|---|---|---|---|---|---|---|---|
| Pass | Size | East | North | RL | Minimum | Maximum | Minimum | Minimum | Maximum |
| 1 | 5x5 | 60 | 60 | 10 | 4 | 16 | 2 | 2 | 4 |
| 2 | 5x5 | 100 | 100 | 10 | 4 | 16 | 1 | 1 | 4 |
| 1 | 15x15 | 125 | 125 | 10 | 4 | 16 | 2 | 2 | 4 |
| 2 | 15x15 | 200 | 200 | 10 | 4 | 16 | 1 | 1 | 4 |
| 1 | 30x30 | 250 | 250 | 10 | 4 | 16 | 2 | 2 | 4 |
| 2 | Unfilled | 500 | 500 | 20 | 1 | 16 | 1 | 1 | 4 |

### 14.2.13 High grade cobalt domain modelling

A high grade Cobalt domain was defined within the GZ domain using an Indicator Modelling approach. Cobalt values above 0.15% were set to 1 and values below 0.15% were set to 0 and this indicator was modelled using anisotropic Inverse Distance Squared ($ID^2$) with the same search parameters as previously described. Three-dimensional shells for an indicator value above 0.5 were generated and the data and blocks inside and outside this shell were separately flagged and modelled. The rationale behind this review was based on concerns about local variability of Cobalt, and to assess if this methodology would prevent potential smearing and smoothing of grades within the generally high-Co GZ domain.

Review of the spatial location of the high grade Co shells with respect to the drill data led to the conclusion that this was a valid and useful tool, and this approach was adopted for the final model.

Typical views of the high grade Co shells are illustrated below.

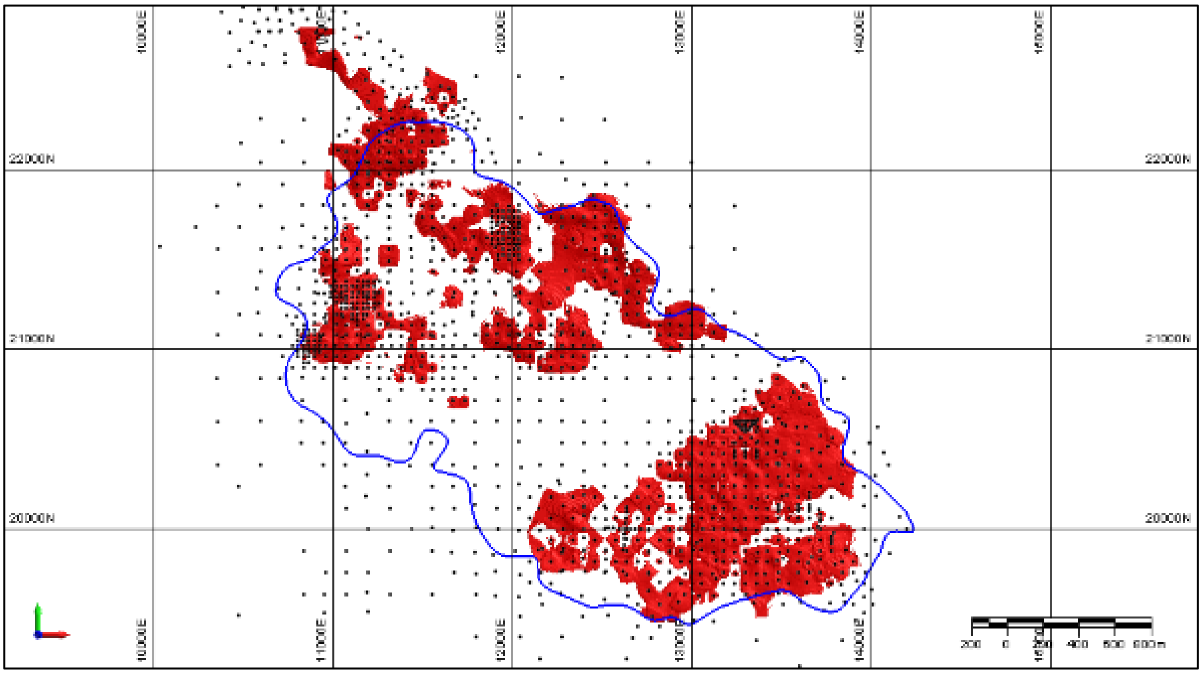



Figure 14-92: High grade cobalt plan view

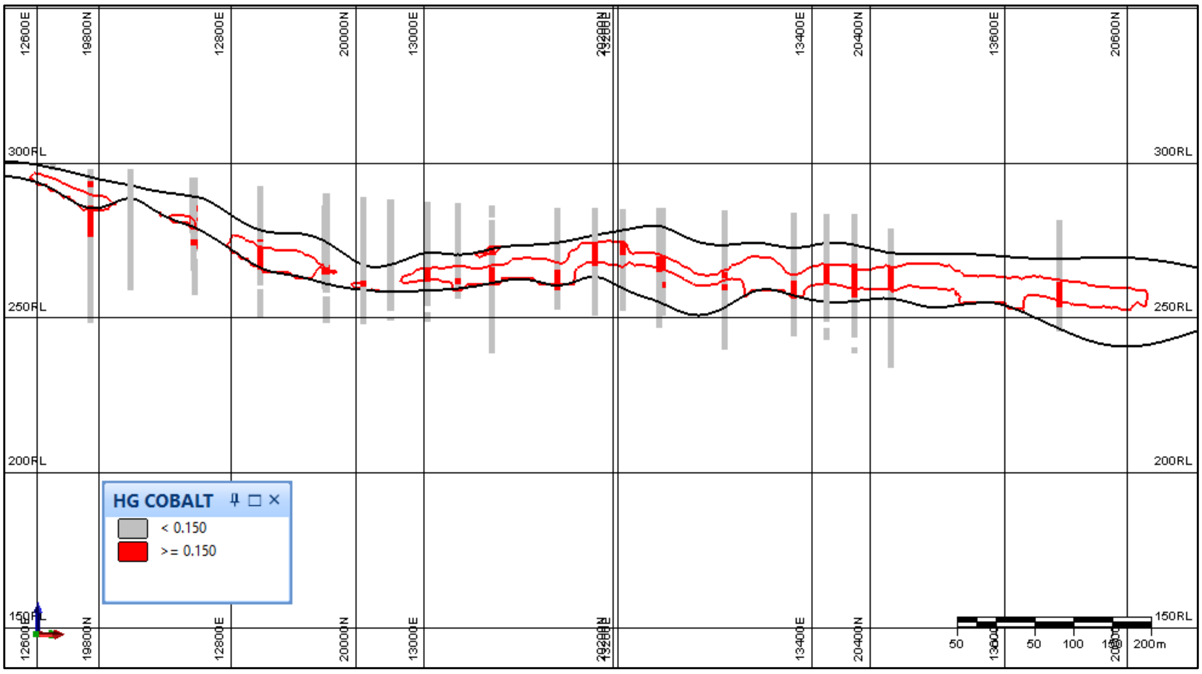



Figure 14-93: Typical section of high grade cobalt zone

## 14.2.14 High grade scandium modelling

As noted in the statistical analysis section, Scandium displays mixed populations both inside and outside the Dunite domain, and within the laterite zone boundaries. Consequently, a similar indicator modelling approach to Co has been taken, but restricted to areas outside the Dunite complex footprint. A threshold of 160 ppm Sc is used to define “high grade scandium”, and examples of the resultant domains are shown below.

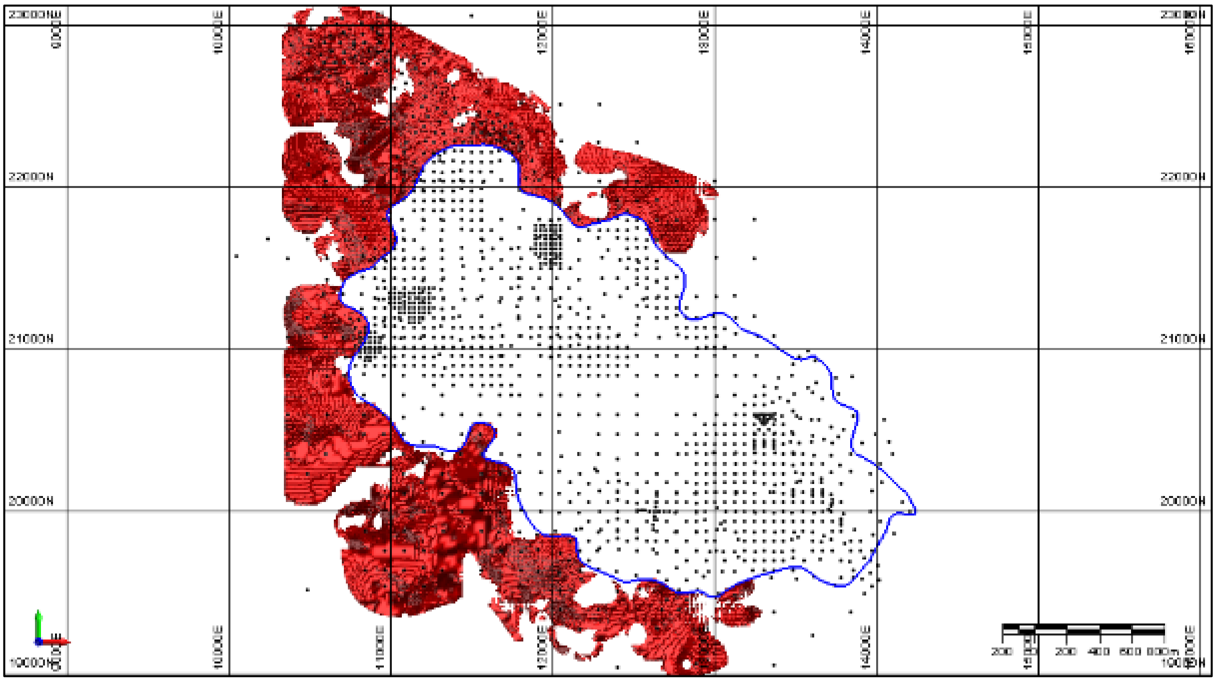


**Figure 14-94: High grade scandium domains**

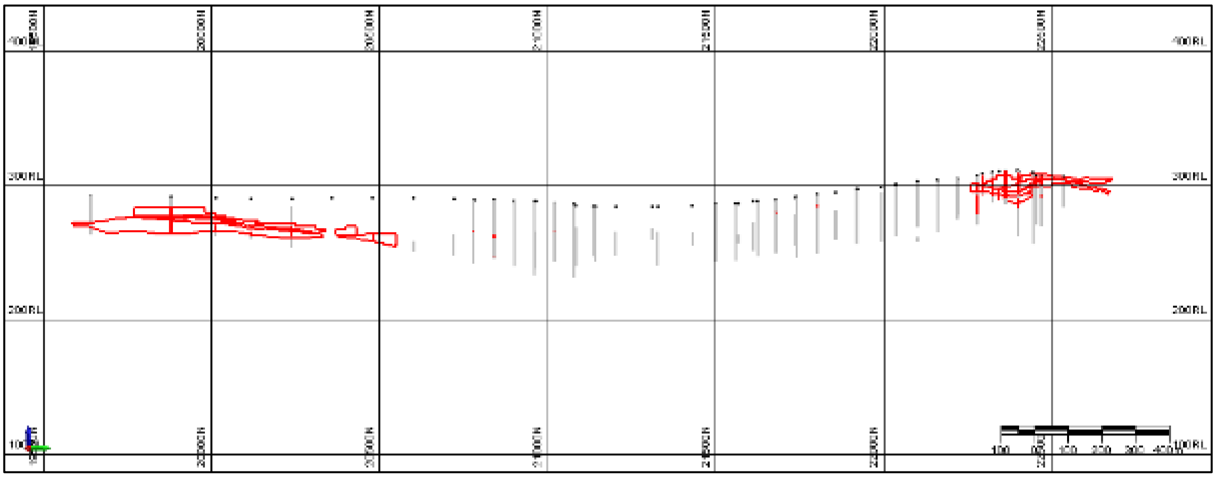


**Figure 14-95: Typical section of high grade scandium zones**

## 14.2.15 Density

Dry bulk density factors used for previous Mineral Resource estimates have been used for this update.

In-situ bulk densities have been determined by measurements carried out on core, measurements at external laboratories and down-hole geophysical logging (gamma-gamma).

The measurements carried out on core were obtained by weighing total material recovered from over 100 m of drilling in mineralised zones by 6 large diameter Calweld holes, adjusted for moisture content determined by oven drying quickly sealed grab samples. As documented, the procedures used seemed appropriate. Due to the relatively large volumes involved these should have been the most reliable measurements available.

Measurements made after drying small core samples from 5 diamond drill holes were given some influence.

Factors applied to the more mineralised zones tended to be slightly rounded downwards. This was prudent in view of the general tendency for a negative correlation between bulk density and grade.

A higher average value was assumed for the SGZ than indicated by the Calweld holes. This was reasonable because they failed to fully penetrate the zone and we would expect average density to increase in its lowermost parts.

Density determination by down-hole geophysical logging were conducted in a total of seven diamond drill holes and about 137 RC holes by either Down Hole Surveys Pty Ltd or Surtron Technologies Pty Ltd in 1999.

Bulk density was assigned by geological domain as tabulated below:

**Table 14-20: Density assigned in model**

| Domain | Code | Density |
|---|---|---|
| Alluvials | AV | 1.80 |
| Overburden | OVB | 1.80 |
| Transition Zone | TZ | 1.70 |
| Goethitic Zone | GZ | 1.20 |
| Silicifed Goethitic Zone | SGZ | 1.25 |
| Saprolite | SAP | 2.00 |

### 14.2.16 Resource model validation

Block model validation has been carried out by several methods, including Swathe Plots, Model versus Data Statistics and Graphs and by Domain and Drill Hole Plan, Section and 3-D Review.

All methods of validation are considered to be acceptable.

Swathe plots for Co and Ni are illustrated below.

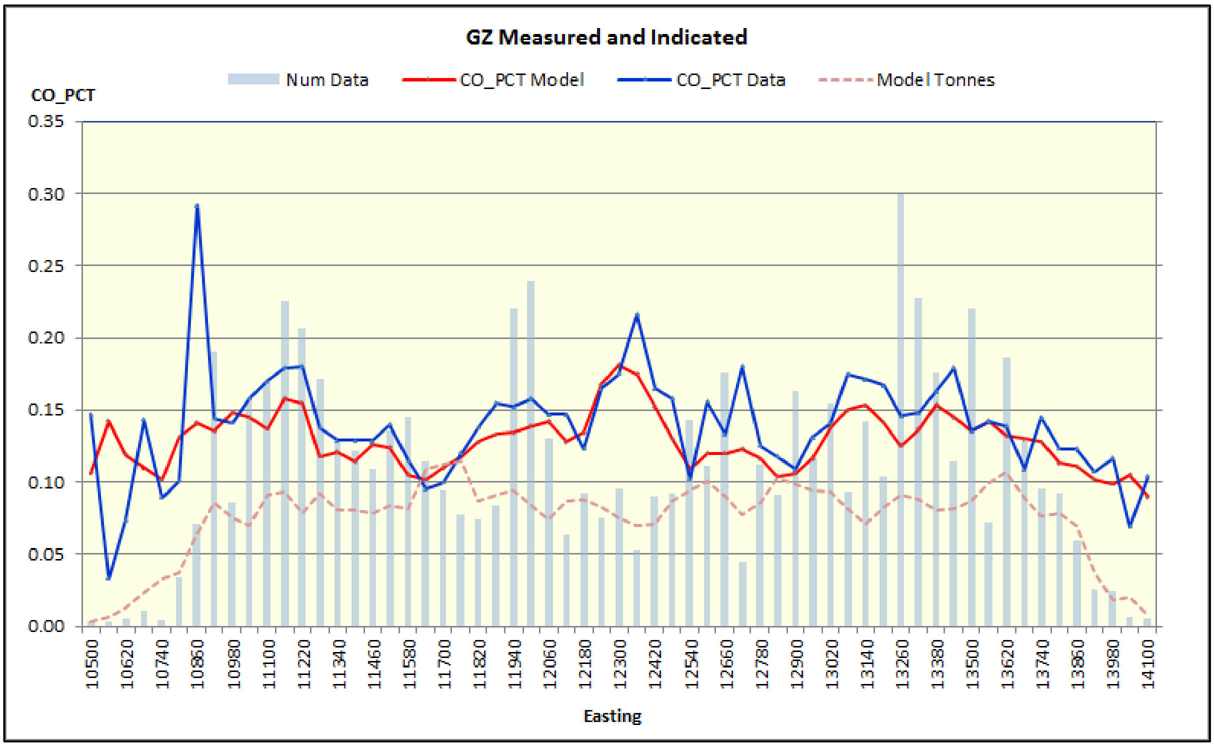



**Figure 14-96: GZ Co by Easting**

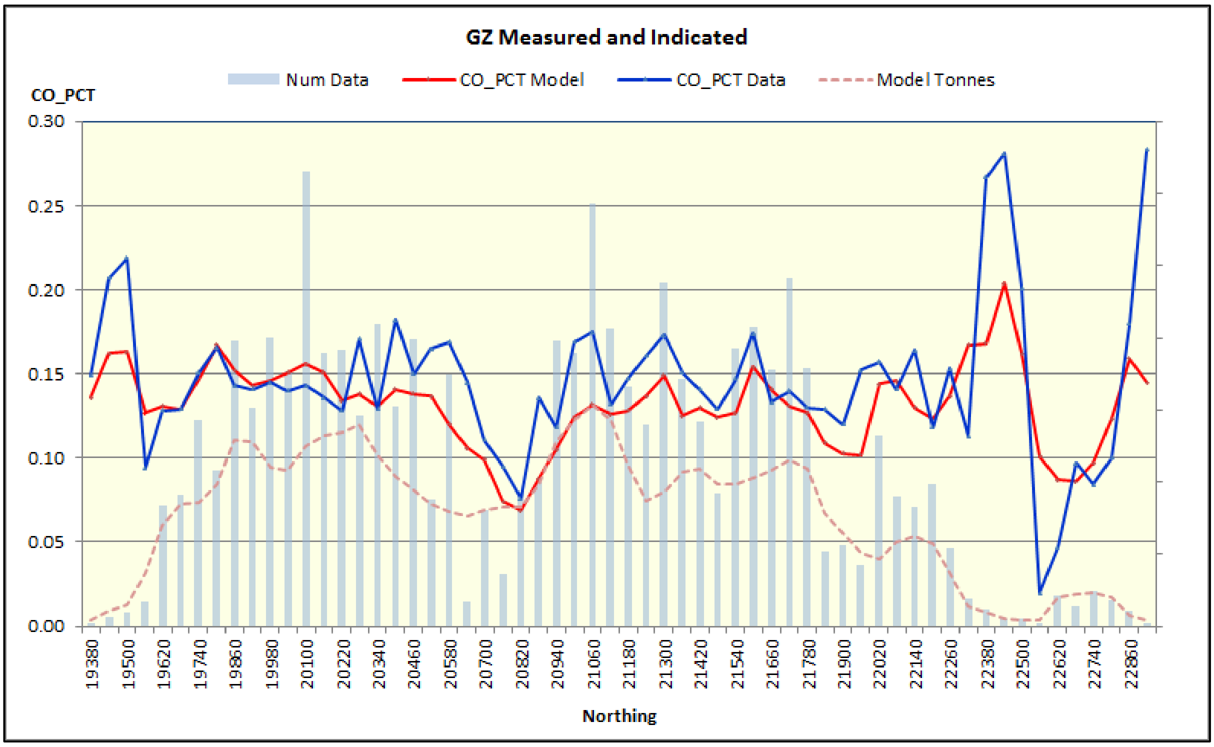



**Figure 14-97: GZ Co by Northing**

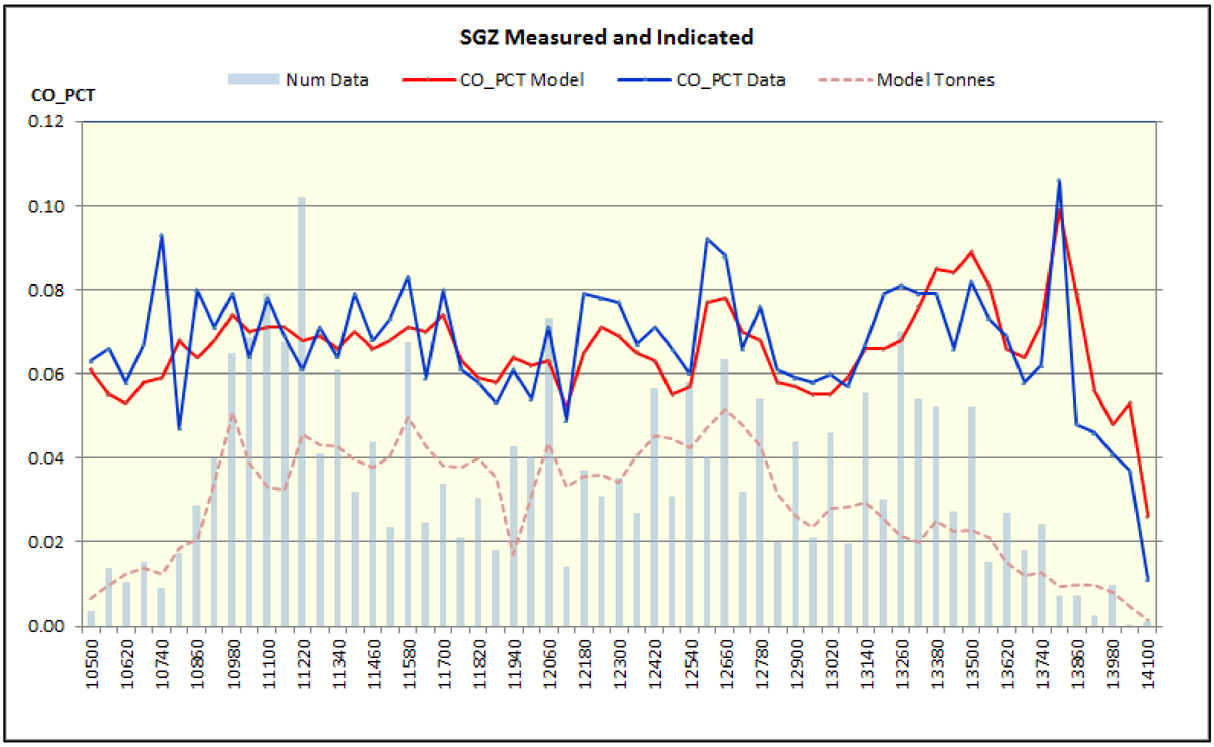



**Figure 14-98: SGZ Co by Easting**

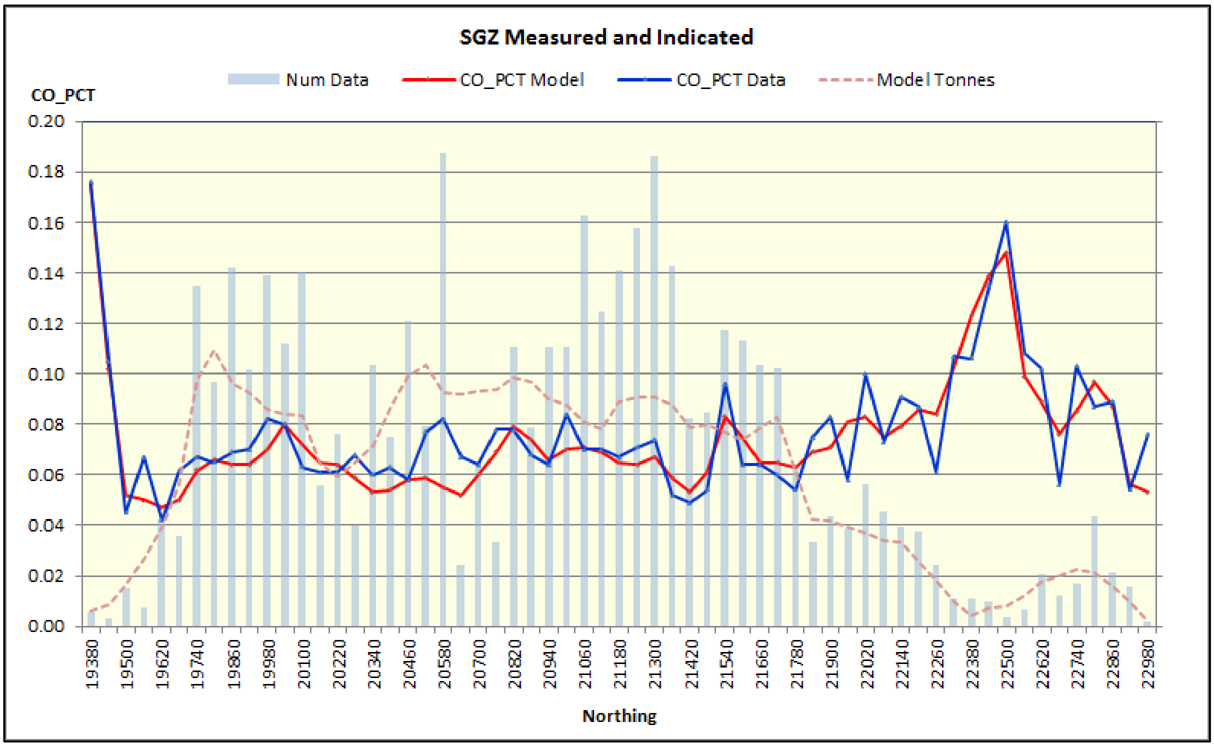



**Figure 14-99: SGZ Co by Northing**

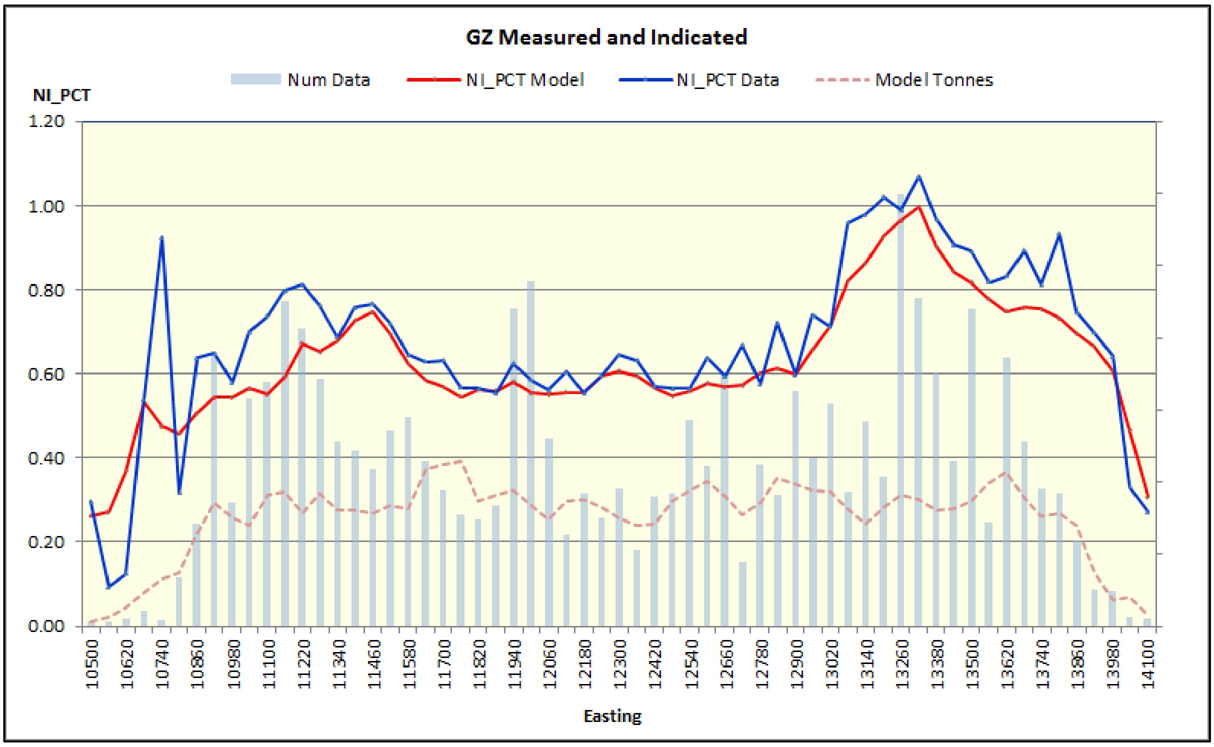



**Figure 14-100: GZ Ni by Easting**

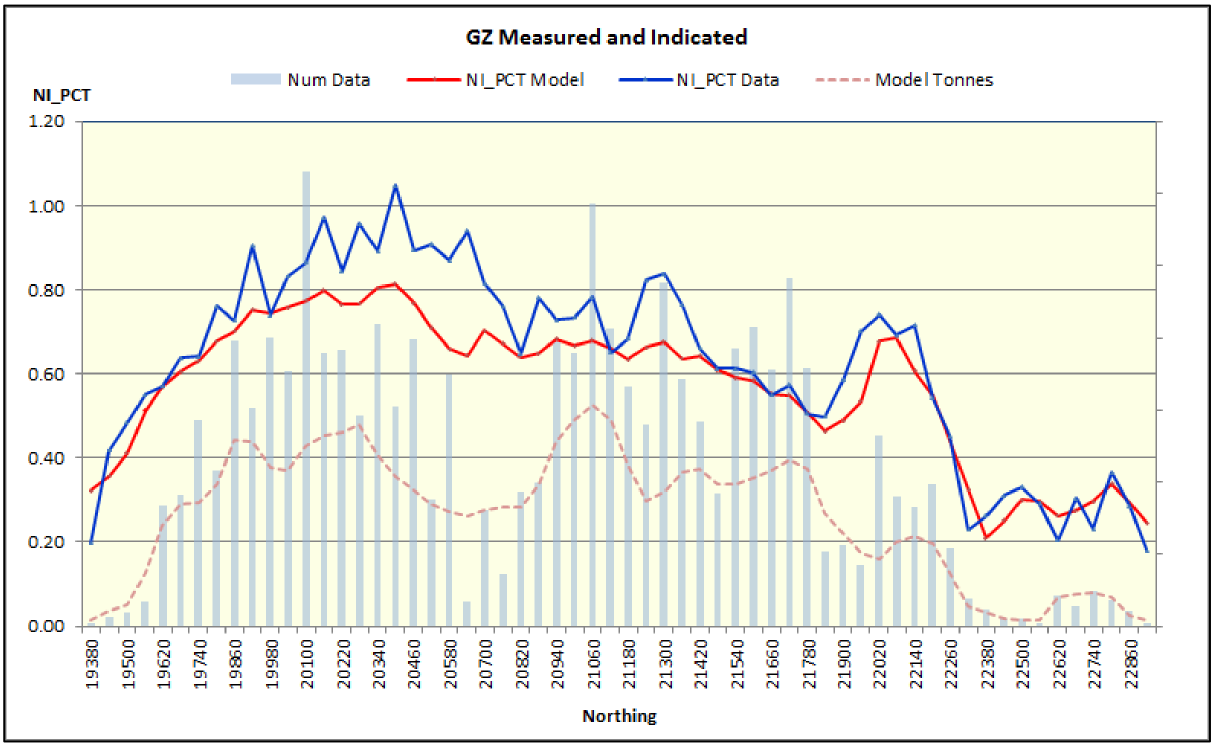



**Figure 14-101: GZ Ni by Easting**

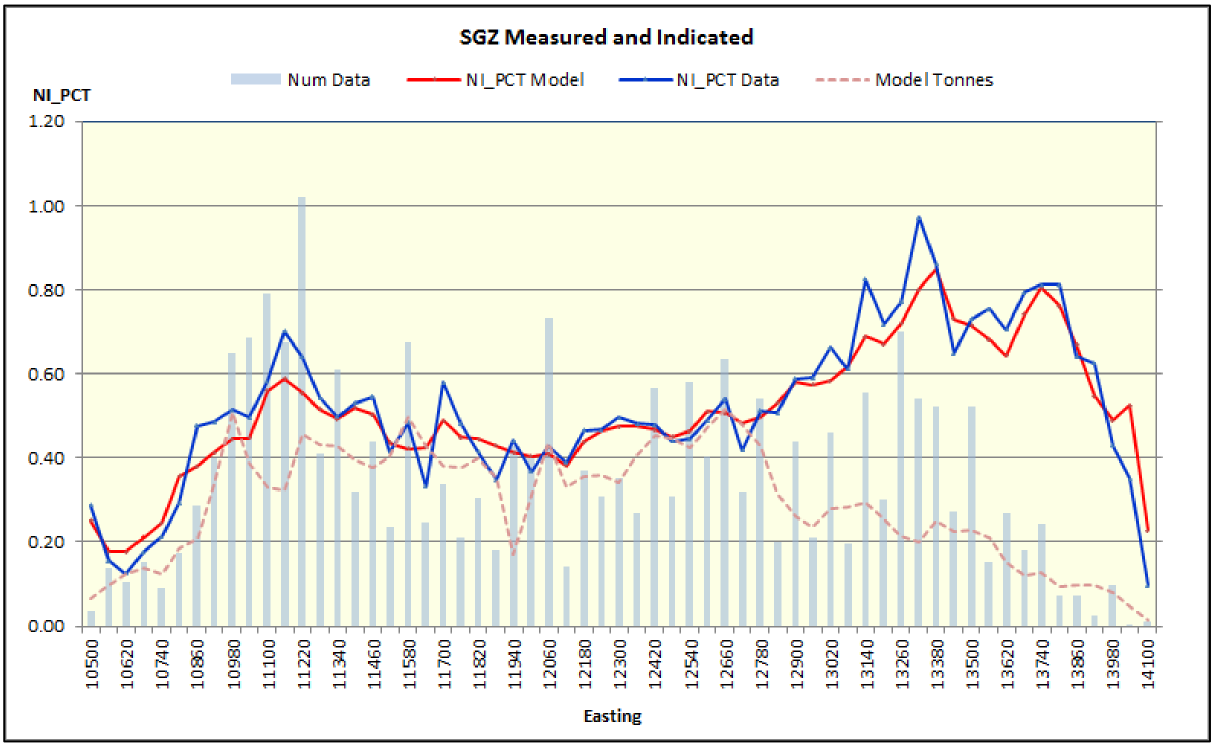



**Figure 14-102: SGZ Ni by Easting**

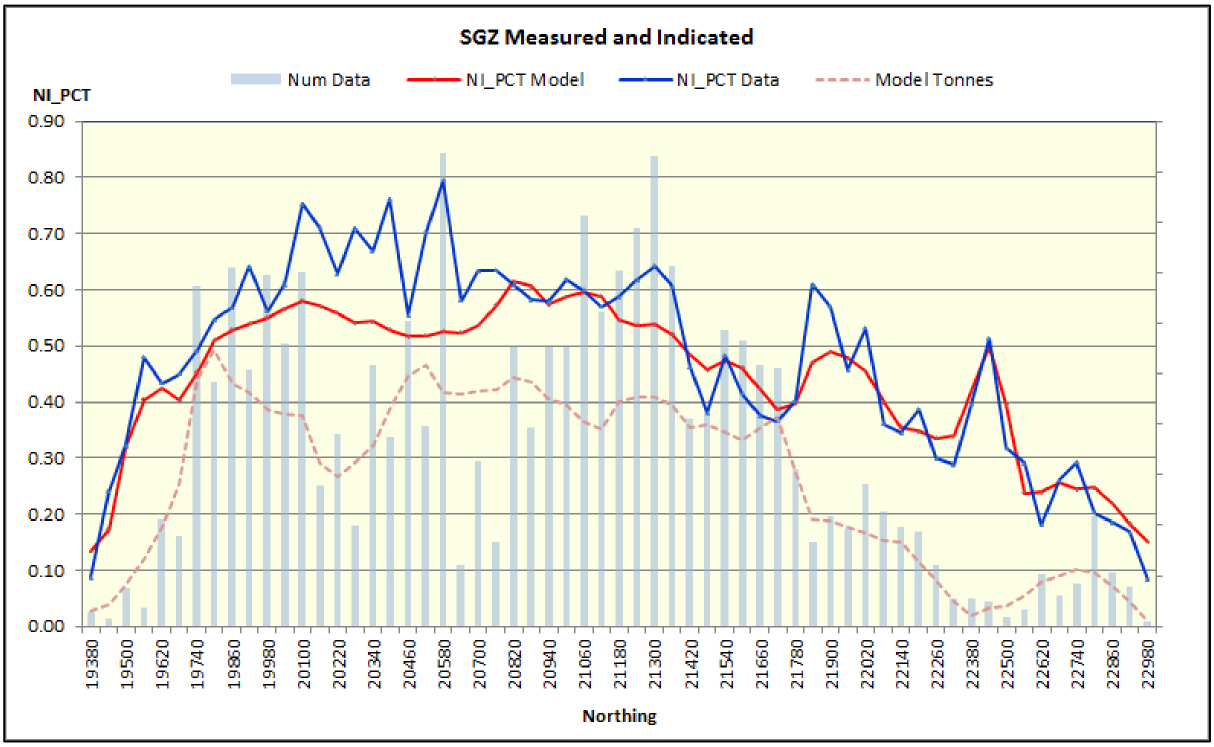



**Figure 14-103: SGZ Ni by Northing**

Declustered data vs model correlation plots are shown below.

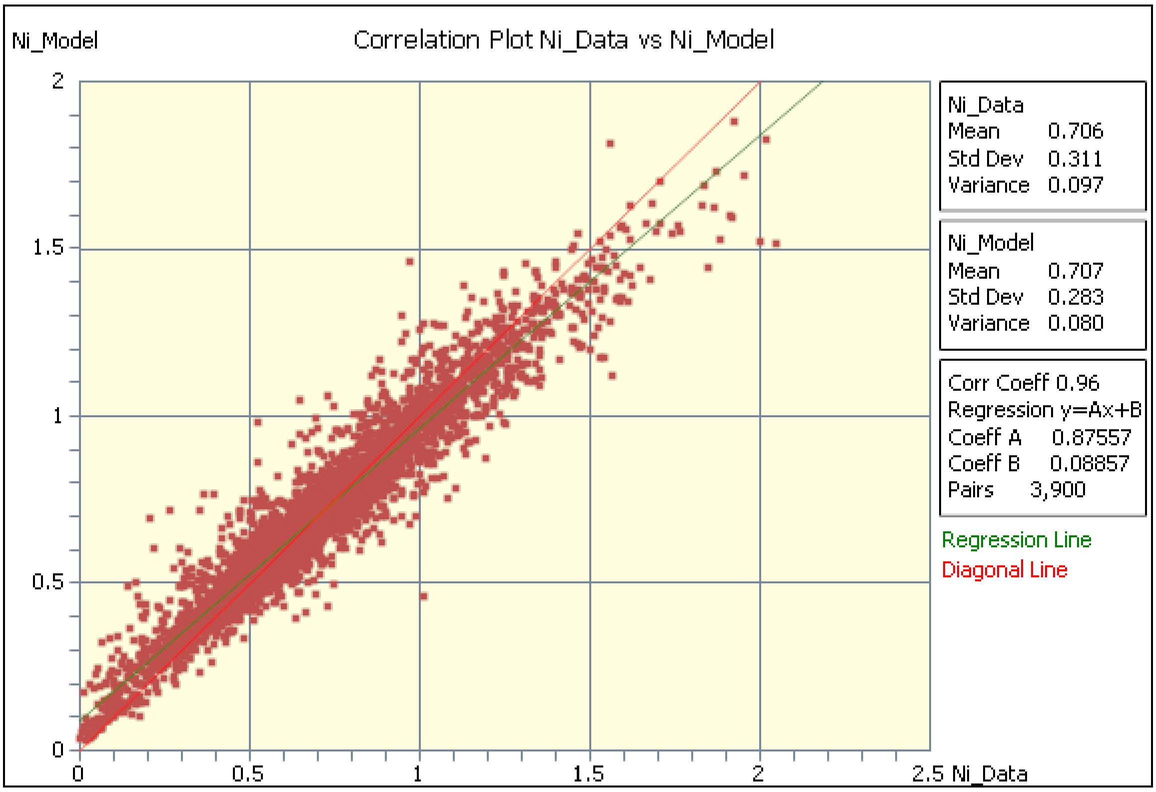



**Figure 14-104: Ni GZ**

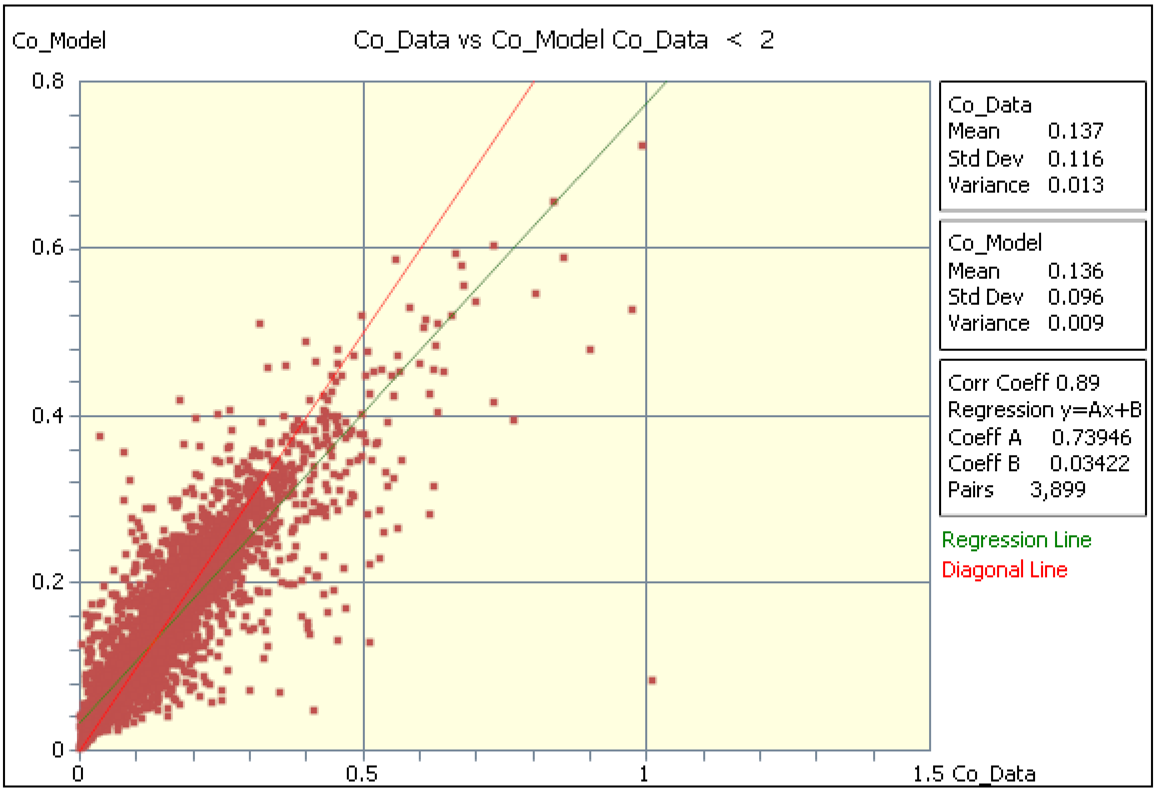



**Figure 14-105: Co GZ**

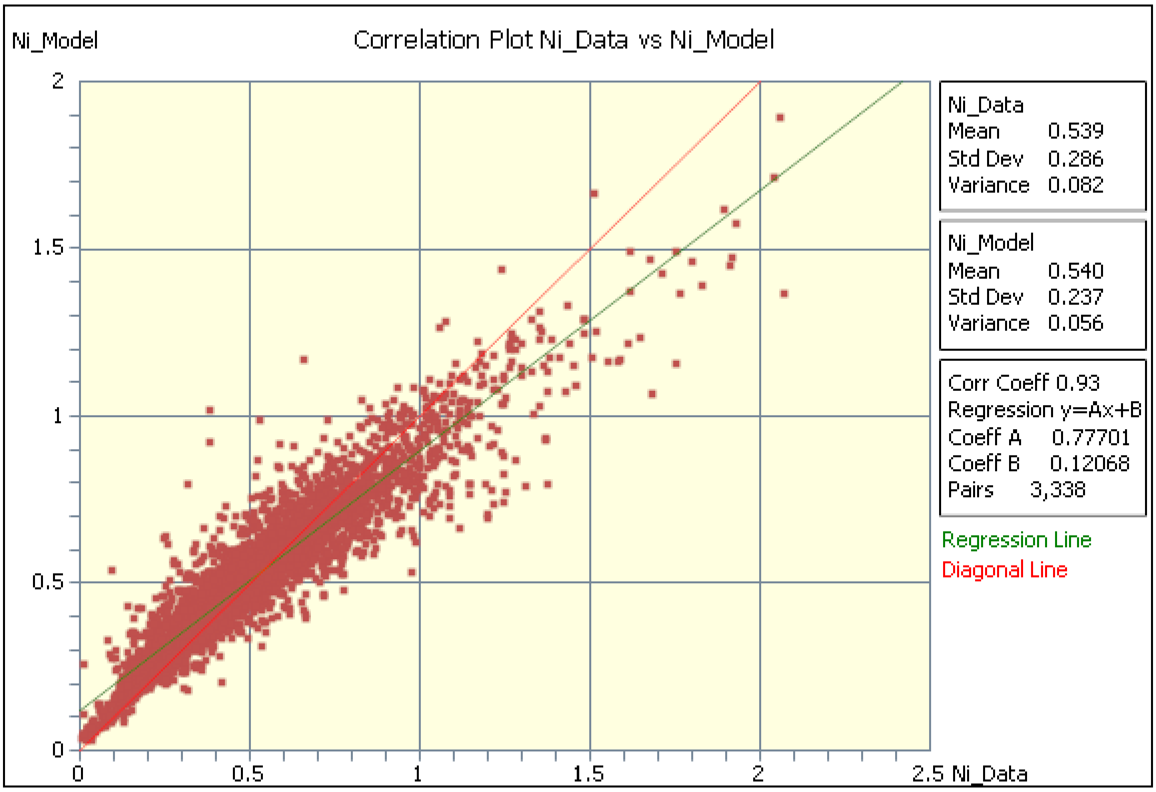



**Figure 14-106: Ni SGZ**

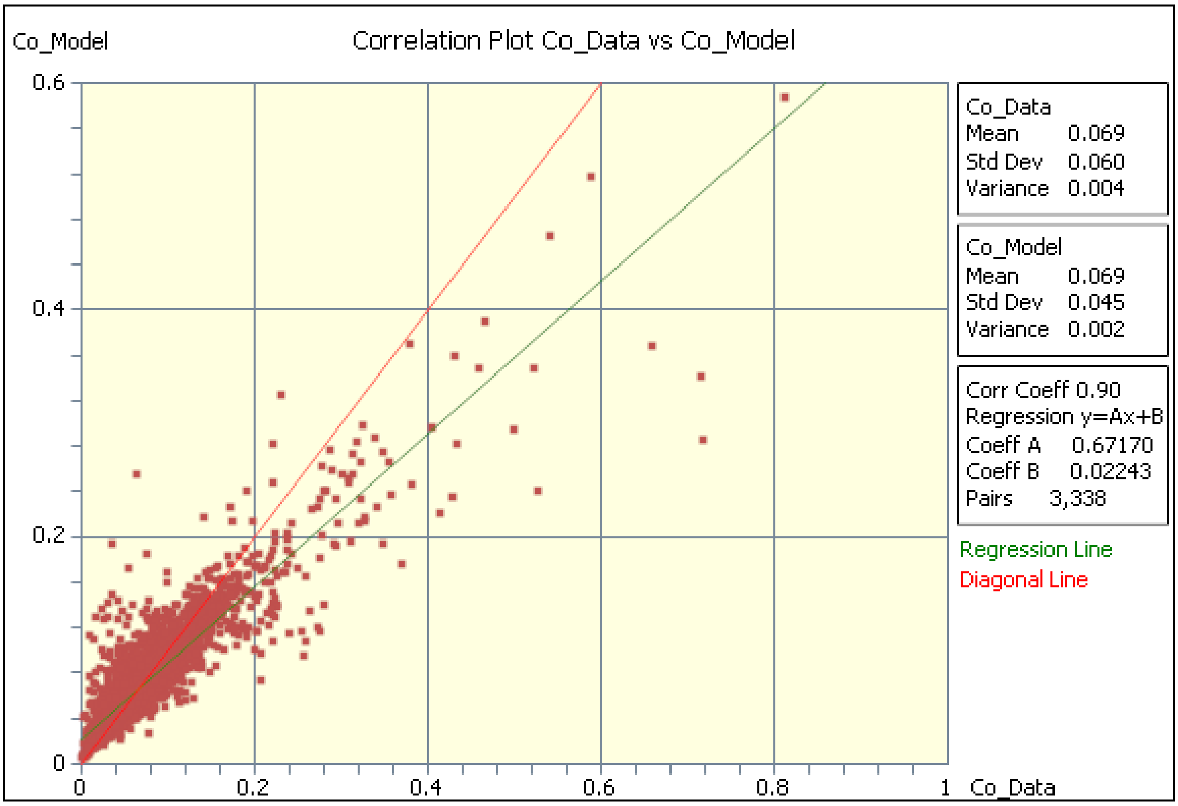



**Figure 14-107: Co SGZ**

## 14.2.17 Resource classification

The Mineral Resource has been classified in the Measured, Indicated and Inferred categories, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code). This is effectively equivalent to the CIM Definition Standards 2014.

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

- Geological continuity
- Data quality
- Drill hole spacing
- Modelling technique
- Estimation properties including search strategy, number of informing data, average distance of data from blocks, kriging variance and kriging efficiency.

The geological continuity of laterite mineralisation is considered to be reasonably well understood. The laterite units have been completely re-interpreted and are now clearly defined in terms of geochemistry and vertical location. The final wireframe surfaces are consistent with the data and spatially consistent, with no overlaps or errors.

Data collection, logging and QAQC of the various drill programs is varied in quality, but validation and verification has been rigorous, with the creation of a new CleanTeq database from raw laboratory data, and rejection of invalid data and holes. Old historical drill holes with less reliable assay and QAQC data have been excluded from the interpolation, though they may sometimes be used to aid geological interpretation.

Drill spacing is one of the major drivers of the resource classification, along with the kriging output parameters. As a general rule, the following spacings characterise the resource classification.

- Nominal 60 m x 60 m or closer Measured
- 120 m x 120 m down to 60 m x 60 m Indicated
- Greater than 120 m x 120 m Inferred.

Estimation of major elements has been by Ordinary Kriging (OK). The interpolation has been set up using Kriging Neighbourhood Analysis and testing of the results of various search and kriging parameter combinations. OK is considered a standard and acceptable estimation methodology for this type of deposit.

The Ordinary Kriging software used (Micromine 2016.1) produces various outputs which can be used as aids in defining resource classification. In this case the following output have been generated for each block:

- Number of composites in the estimation of each block
- Search pass number (generally three search passes are used to ensure full population of the block model)
- Number of holes in the estimation of each block
- Average distance of composites from block centre
- Kriging variance
- Kriging efficiency
- Slope of regression.

All of the above, along with the drill spacing and confidence in the geological interpretation, are used to arrive at the final resource classification.

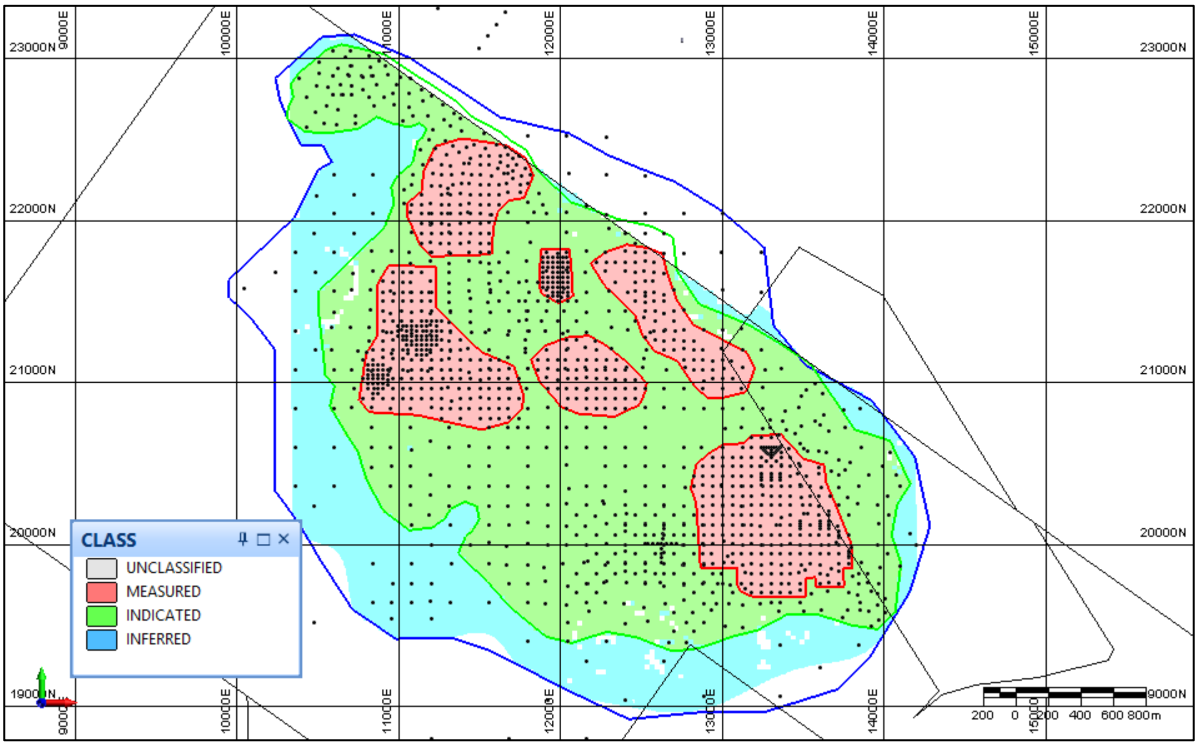



**Figure 14-108: Resource classification boundaries**

Scandium and platinum are considered to be potentially economically recoverable elements, but have not been assayed as consistently in the raw data as the cobalt, nickel and most other elements. Consequently, different resource classification schemes have been derived for each of these elements.

Scandium uses the same classification criteria as Cobalt for the TZ, GZ, SGZ and SAP zones, but due to limited assaying in AV and OVB, a different classification is used in these zones, as shown below.

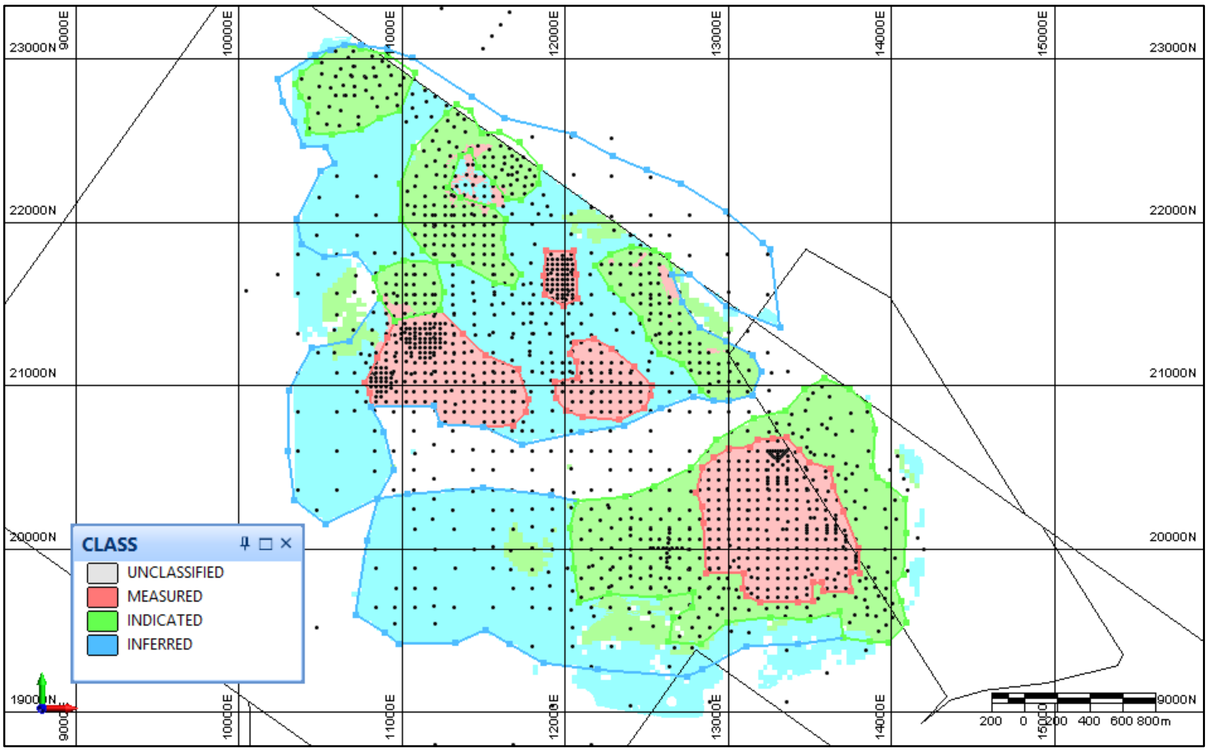



**Figure 14-109: Scandium Resource classification**

The Platinum data is more limited in all domains and a different set of classification areas are used.

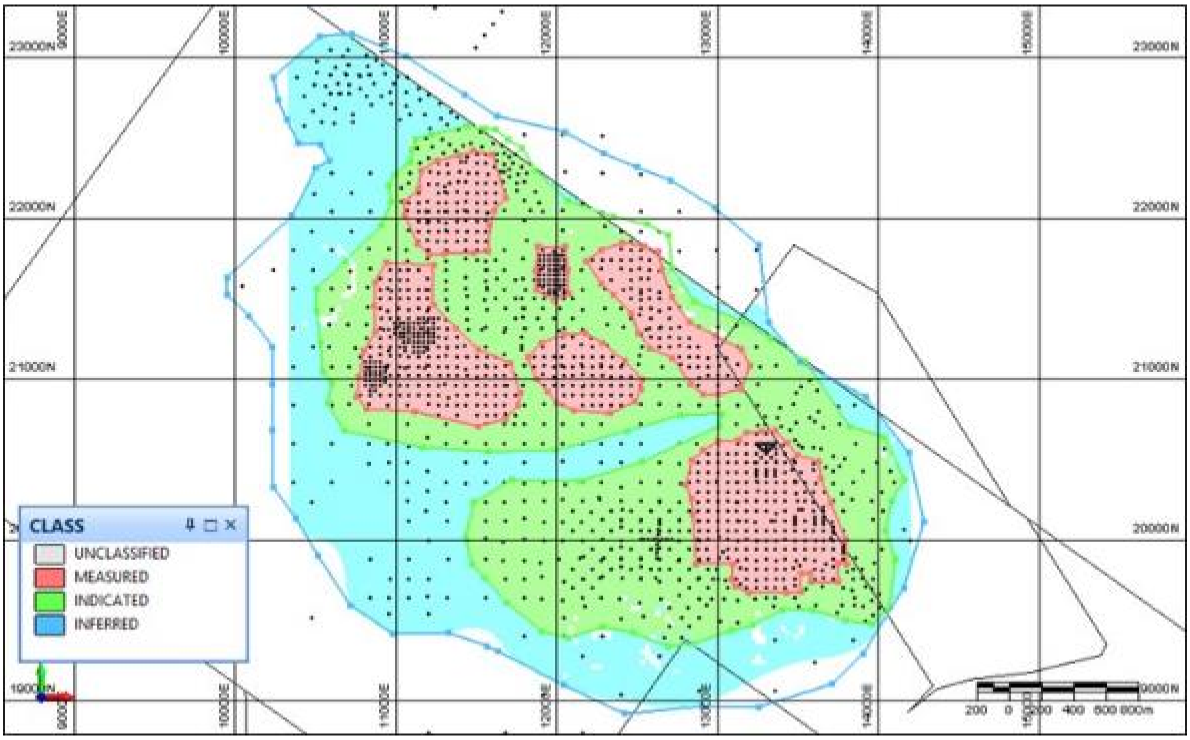



**Figure 14-110: Platinum Resource classification**

A final adjustment is made to the Saprolite classification for all blocks. Since there is limited data in the SAP domain, and there is no reliable base of Saprolite, Indicated SAP is downgraded to Inferred and Measured SAP is downgraded to Indicated.

### 14.2.18 Cut-off grade & nickel equivalent factors

Nickel-equivalent factors have not been used to report the resource as it is now considered primarily to be a Cobalt resource. Separate cut-offs have been used reporting separate resources of Cobalt (with associated Nickel), Scandium and Platinum.

### 14.2.19 Mining assumptions

Due to the proximity of the mineralisation to surface, the deposit is amenable to conventional open pit mining. Two feasibility studies have developed practicable staged open pit mine plans based on conventional open pit mining by contractor, using large backhoes and trucks, operating on working benches 2m in height. The most recent study assumed about 2.5 Mtpa of feed to a processing plant.

No dilution or ore loss is specifically included in the resource model, other than that inherent in the smoothing introduced by the kriging interpolation methodology and the inherent dilution built into the geological modelling as precursor to the Resource Modelling and Estimation.

### 14.2.20 Metallurgical assumptions

Metallurgical test work has been carried out on diamond, reverse circulation and sonic core samples from geographically dispersed drill holes, with coverage of all geological domains.

Metallurgical Test work on the Nickel, Cobalt and Platinum material for the Syerston project was completed by Black Range Minerals and Ivanplats, through ALS Metallurgy, SGS Metallurgy, Hazen Laboratories and other laboratories as part of the feasibility studies conducted in 2000 and 2005.

Additional test work, including Pilot Scale test work, was carried out on the Nickel, Cobalt and Scandium material by ALS Metallurgy, SGS Metallurgy and other laboratories during a Feasibility Study (FS) in 2016 and 2017 for mineral recovery determination.

A comprehensive suite of metallurgical test work, including further Pilot Scale test work and specific equipment vendor test work is planned as part of the Definitive Feasibility Study (DFS), currently being undertaken by Clean TeQ. The ongoing metallurgical test work shall include metallurgical samples and composites collected from bulk test pits and geographically dispersed drill holes.

Average overall metallurgical recoveries to final product were estimated to be 90.0% for Ni and 88.9% for Co. The metallurgical recoveries for Ni and Co were derived from metallurgical test work comprising over 150 ore variability batch tests and 3 pilot plant campaigns testing 6 ore composites as part of 3 feasibility studies completed in 2000, 2005 and 2017. Recent metallurgical test work undertaken by Clean TeQ confirm these recoveries.

Results of average feed grades support resource grades

Sufficient work has been undertaken to demonstrate that a potentially viable treatment process is available for the Syerston lateritic Ni, Co and Sc mineralisation. The proposed process for Nickel and Cobalt recovery involves high pressure acid leaching followed by continuous RIP process for the extraction of Nickel, Cobalt and Scandium from solution which is then purified via separation of Scandium via ion exchange followed by and solvent extraction separation and purification to prior to crystallisation to produce battery grade Nickel and Cobalt sulphates. The proposed process for the Scandium recovery involves precipitation and purification steps of the Scandium eluate to produce high purity Scandium oxide product.

### 14.2.21 Environmental assumptions

The area in which the deposit occurs does not seem to have any unusual environmental significance.

An Environmental Impact Statement (EIS) was prepared in parallel with the 2000 feasibility study and in May 2001 the proposed Ni-Co project received Development Consent under the NSW Environmental Planning and Assessment Act.

The previous granting of a Development Consent indicates that there are unlikely to be any insurmountable environmental obstacles.

Additional permits and licences would have to be obtained before operations could commence.

As part of the DFS, additional baseline studies are being undertaken to assess potential environmental impacts of the mining and processing operations.

There are no obvious environmental factors that would prevent the deposit being reported as an identified mineral resource.

### 14.2.22 2017 Resource estimate

The cobalt grade of the Mineral Resource has increased by 30%. The Mineral Resource is now 101 million tonnes at 0.13% Co for contained cobalt metal of 132,000 t. The nickel grade of the resource is 0.59% Ni for 593,000 t of contained nickel. Of this total resource, 86% is in the Measured and Indicated categories.

The updated cobalt and nickel Mineral Resource is summarised below at a 0.06% Co cut-off grade. and also at range of Cobalt cut-off grades.

**Table 14-21: Syerston cobalt/ nickel Mineral Resource estimate at 0.06% Co cut-off**

| Classification | Tonnes | Co Grade | Ni Grade | Co Metal | Ni Metal |
|---|---|---|---|---|---|
| **Category** | **Millions** | **%** | **%** | **Tonnes** | **Tonnes** |
| Measured | 40 | 0.15 | 0.75 | 59,000 | 298,942 |
| Indicated | 47 | 0.12 | 0.55 | 58,000 | 259,479 |
| **Measures + Indicated** | **87** | **0.13** | **0.64** | **116,000** | **558,421** |
| Inferred | 14 | 0.11 | 0.24 | 16,000 | 34,643 |
| **Total** | **101** | **0.13** | **0.59** | **132,000** | **593,070** |
| *Rounding errors may occur* | | | | | |

**Table 14-22: Syerston cobalt/ nickel Mineral Resource estimate at a 0.0% Co cut-off**

| Classification | Tonnes | Co Grade | Ni Grade | Co Metal | Ni Metal |
|---|---|---|---|---|---|
| **Category** | **Millions** | **%** | **%** | **Tonnes** | **Tonnes** |
| Measured | 69 | 0.10 | 0.63 | 69,000 | 436,000 |
| Indicated | 94 | 0.08 | 0.47 | 75,000 | 438,000 |
| **Measures + Indicated** | **163** | **0.09** | **0.54** | **144,000** | **874,000** |
| Inferred | 21 | 0.09 | 0.23 | 18,000 | 48,000 |
| **Total** | **183** | **0.09** | **0.50** | **162,000** | **922,000** |
| *Rounding errors may occur* | | | | | |

**Table 14-23 Syerston cobalt/ nickel Mineral Resource estimate at a 0.08% Co cut-off**

| Classification | Tonnes | Co Grade | Ni Grade | Co Metal | Ni Metal |
|---|---|---|---|---|---|
| **Category** | **Millions** | **%** | **%** | **Tonnes** | **Tonnes** |
| Measured | 29 | 0.18 | 0.78 | 51,000 | 227,000 |
| Indicated | 32 | 0.15 | 0.57 | 47,000 | 183,000 |
| **Measures + Indicated** | **61** | **0.16** | **0.67** | **98,000** | **410,000** |
| Inferred | 10 | 0.13 | 0.25 | 13,000 | 25,000 |
| **Total** | **71** | **0.16** | **0.61** | **111,000** | **435,000** |
| *Rounding errors may occur* | | | | | |

**Table 14-24 Syerston cobalt/ nickel Mineral Resource estimate at a 0.10% Co cut-off**

| Classification | Tonnes | Co Grade | Ni Grade | Co Metal | Ni Metal |
|---|---|---|---|---|---|
| **Category** | **Millions** | **%** | **%** | **Tonnes** | **Tonnes** |
| Measured | 22 | 0.20 | 0.80 | 44,000 | 175,000 |
| Indicated | 21 | 0.18 | 0.59 | 38,000 | 126,000 |
| **Measures + Indicated** | **43** | **0.19** | **0.70** | **82,000** | **302,000** |
| Inferred | 7 | 0.15 | 0.25 | 10,000 | 17,000 |
| **Total** | **50** | **0.19** | **0.64** | **93,000** | **318,000** |
| *Rounding errors may occur* | | | | | |

The Scandium and Platinum Resources are reported below.

**Table 14-25 Syerston scandium Mineral Resource estimate (300 ppm Sc cut-off)**

| | | Mt | Sc ppm | Sc metal | Sc2O3 |
|---|---|---|---|---|---|
| Alluvial | Measured | | | | |
| | Indicated | 1.12 | 368 | 411 | 629 |
| | Inferred | 7.29 | 366 | 2,671 | 4,086 |
| Overburden | Measured | 0.01 | 348 | 2 | 4 |
| | Indicated | 1.29 | 395 | 511 | 781 |
| | Inferred | 17.01 | 421 | 7,158 | 10,952 |
| Transition | Measured | 0.40 | 434 | 174 | 266 |
| | Indicated | 2.28 | 414 | 945 | 1,446 |
| | Inferred | 1.87 | 446 | 833 | 1,274 |
| Goethite | Measured | 0.91 | 512 | 467 | 714 |
| | Indicated | 2.82 | 443 | 1,251 | 1,914 |
| | Inferred | 3.98 | 536 | 2,133 | 3,263 |
| Sliceous Goethite | Measured | 0.44 | 439 | 191 | 293 |
| | Indicated | 3.05 | 401 | 1,223 | 1,871 |
| | Inferred | 3.19 | 392 | 1,252 | 1,916 |
| Total | Measured+Indicated | 12.32 | 420 | 5,175 | 7,918 |
| | Inferred | 33.34 | 421 | 14,047 | 21,492 |
| | Total | 45.66 | 421 | 19,222 | 29,409 |

**Table 14-26: Syerston scandium Mineral Resource estimate Breakdown by Location**

| Domain | Tonnes Millions | Sc Ppm | Sc Metal Tonnes | $Sc_2O_3$ Equiv Tonnes |
|---|---|---|---|---|
| Within Dunite Complex (above 300 ppm) | 0.3 | 343 | 100 | 153 |
| Outside Dunite Complex (above 300 ppm) | 45.4 | 421 | 19,122 | 29,256 |
| **Total** | **45.7** | **421** | **19,222** | **29,409** |

**Table 14-27: Platinum Mineral Resource estimate**

| Pt Cutoff | Class | Mt | Pt g/t | Ounces |
|---|---|---|---|---|
| 0.15 | Measured | 36.68 | 0.37 | 431,491 |
| | Indicated | 52.33 | 0.30 | 509,065 |
| | Measured+Indicated | 89.01 | 0.33 | 940,557 |
| | Inferred | 14.08 | 0.30 | 135,614 |
| | Total | 103.09 | 0.32 | 1,076,170 |
| 0.50 | Measured | 5.33 | 0.94 | 161,773 |
| | Indicated | 5.16 | 0.70 | 116,792 |
| | Measured+Indicated | 10.49 | 0.83 | 278,565 |
| | Inferred | 1.65 | 0.79 | 41,713 |
| | Total | 12.15 | 0.82 | 320,278 |
| 1.00 | Measured | 1.06 | 2.12 | 72,507 |
| | Indicated | 0.43 | 1.47 | 20,269 |
| | Measured+Indicated | 1.49 | 1.93 | 92,776 |
| | Inferred | 0.23 | 1.44 | 10,745 |
| | Total | 1.73 | 1.87 | 103,521 |

The updated platinum Resource is inclusive of a higher grade zone of 1.7 Mt @ 1.87 g/t Pt for 103,435 ounces at a 1 g/t Pt cut-off grade.

**Table 14-28: Syerston nickel/ cobalt Mineral Resource estimate by domain (0.06% Co cut-off)**

| Latzone | Class | Mt | Ni % | Co % | Fe % | Al % | Si % | Mg % | Mn % | Ca % | Cu ppm | Zn ppm | Cr ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Transition | Measured | 1.16 | 0.42 | 0.10 | 38.56 | 5.12 | 8.14 | 0.77 | 0.60 | 0.48 | 81 | 269 | 6,610 |
| | Indicated | 2.42 | 0.30 | 0.10 | 38.13 | 5.46 | 7.97 | 0.42 | 0.57 | 0.27 | 111 | 282 | 5,452 |
| | M+I | 3.57 | 0.34 | 0.10 | 38.27 | 5.35 | 8.03 | 0.53 | 0.58 | 0.33 | 101 | 278 | 5,827 |
| | Inferred | 1.96 | 0.24 | 0.09 | 37.50 | 5.51 | 8.42 | 0.33 | 0.55 | 0.17 | 86 | 274 | 4,564 |
| | Total | 5.53 | 0.30 | 0.10 | 38.00 | 5.41 | 8.18 | 0.46 | 0.57 | 0.28 | 96 | 277 | 5,380 |
| Goethite | Measured | 25.31 | 0.81 | 0.18 | 42.59 | 2.92 | 7.80 | 0.76 | 0.95 | 0.24 | 67 | 411 | 5,530 |
| | Indicated | 26.65 | 0.59 | 0.14 | 41.83 | 3.04 | 8.26 | 0.80 | 0.83 | 0.31 | 98 | 393 | 5,618 |
| | M+I | 51.96 | 0.70 | 0.16 | 42.20 | 2.98 | 8.03 | 0.78 | 0.88 | 0.28 | 83 | 402 | 5,575 |
| | Inferred | 6.97 | 0.25 | 0.13 | 33.72 | 6.36 | 10.48 | 0.76 | 0.82 | 0.36 | 214 | 309 | 3,558 |
| | Total | 58.93 | 0.64 | 0.16 | 41.20 | 3.38 | 8.33 | 0.78 | 0.88 | 0.29 | 98 | 391 | 5,337 |
| Siliceous Goethite | Measured | 13.44 | 0.67 | 0.10 | 27.56 | 1.40 | 22.27 | 0.84 | 0.60 | 0.20 | 31 | 304 | 5,832 |
| | Indicated | 17.96 | 0.53 | 0.09 | 27.11 | 1.78 | 22.59 | 1.02 | 0.57 | 0.29 | 51 | 278 | 5,724 |
| | M+I | 31.40 | 0.59 | 0.10 | 27.30 | 1.62 | 22.45 | 0.94 | 0.58 | 0.25 | 42 | 289 | 5,770 |
| | Inferred | 5.27 | 0.24 | 0.10 | 24.59 | 4.40 | 19.91 | 1.68 | 0.58 | 0.82 | 200 | 339 | 4,471 |
| | Total | 36.67 | 0.54 | 0.10 | 26.91 | 2.02 | 22.08 | 1.05 | 0.58 | 0.33 | 65 | 297 | 5,584 |
| Total | Measured | 39.90 | 0.75 | 0.15 | 37.41 | 2.47 | 12.68 | 0.79 | 0.82 | 0.23 | 56 | 371 | 5,663 |
| | Indicated | 47.03 | 0.55 | 0.12 | 36.02 | 2.69 | 13.74 | 0.87 | 0.72 | 0.30 | 80 | 343 | 5,650 |
| | M+I | 86.93 | 0.64 | 0.13 | 36.66 | 2.59 | 13.25 | 0.83 | 0.76 | 0.27 | 69 | 356 | 5,656 |
| | Inferred | 14.20 | 0.24 | 0.11 | 30.85 | 5.52 | 13.70 | 1.04 | 0.69 | 0.50 | 191 | 315 | 4,035 |
| | Total | 101.13 | 0.59 | 0.13 | 35.84 | 3.00 | 13.31 | 0.86 | 0.75 | 0.30 | 86 | 350 | 5,429 |

**Table 14-29: Syerston nickel and cobalt Mineral Resource estimate by domain (No Co cut-off)**

| Latzone | Class | Mt | Ni % | Co % | Fe % | Al % | Si % | Mg % | Mn % | Ca % | Cu ppm | Zn ppm | Cr ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Transition | Measured | 10.46 | 0.37 | 0.04 | 38.11 | 5.22 | 8.04 | 1.03 | 0.26 | 0.80 | 69 | 233 | 7,329 |
| | Indicated | 16.97 | 0.30 | 0.04 | 38.59 | 5.30 | 7.80 | 0.69 | 0.26 | 0.44 | 90 | 248 | 7,200 |
| | M+I | 27.43 | 0.32 | 0.04 | 38.41 | 5.27 | 7.90 | 0.82 | 0.26 | 0.58 | 82 | 242 | 7,249 |
| | Inferred | 3.82 | 0.22 | 0.06 | 37.03 | 5.81 | 8.40 | 0.63 | 0.44 | 0.52 | 93 | 239 | 4,175 |
| | Total | 31.25 | 0.31 | 0.04 | 38.24 | 5.33 | 7.96 | 0.80 | 0.28 | 0.57 | 84 | 242 | 6,873 |
| Goethite | Measured | 32.24 | 0.76 | 0.15 | 42.51 | 3.05 | 7.71 | 0.78 | 0.84 | 0.30 | 66 | 391 | 5,667 |
| | Indicated | 36.53 | 0.56 | 0.12 | 41.66 | 3.09 | 8.37 | 0.78 | 0.70 | 0.31 | 91 | 370 | 5,902 |
| | M+I | 68.77 | 0.65 | 0.13 | 42.06 | 3.07 | 8.06 | 0.78 | 0.76 | 0.31 | 79 | 380 | 5,792 |
| | Inferred | 8.09 | 0.24 | 0.12 | 33.48 | 6.31 | 10.69 | 0.77 | 0.77 | 0.39 | 214 | 300 | 3,539 |
| | Total | 76.87 | 0.61 | 0.13 | 41.16 | 3.41 | 8.33 | 0.78 | 0.76 | 0.32 | 93 | 372 | 5,555 |
| Siliceous Goethite | Measured | 26.11 | 0.58 | 0.07 | 23.76 | 1.21 | 25.12 | 0.94 | 0.48 | 0.25 | 27 | 249 | 5,278 |
| | Indicated | 40.39 | 0.45 | 0.06 | 23.68 | 1.64 | 24.83 | 1.21 | 0.42 | 0.38 | 43 | 233 | 5,298 |
| | M+I | 66.50 | 0.50 | 0.07 | 23.72 | 1.47 | 24.95 | 1.10 | 0.45 | 0.33 | 37 | 239 | 5,290 |
| | Inferred | 8.72 | 0.23 | 0.08 | 24.31 | 4.19 | 20.32 | 1.86 | 0.50 | 0.87 | 168 | 320 | 3,873 |
| | Total | 75.22 | 0.47 | 0.07 | 23.78 | 1.79 | 24.40 | 1.19 | 0.45 | 0.39 | 52 | 249 | 5,126 |
| Total | Measured | 68.81 | 0.63 | 0.10 | 34.73 | 2.68 | 14.37 | 0.88 | 0.61 | 0.36 | 52 | 313 | 5,772 |
| | Indicated | 93.90 | 0.47 | 0.08 | 33.37 | 2.86 | 15.39 | 0.95 | 0.50 | 0.36 | 70 | 289 | 5,877 |
| | M+I | 162.70 | 0.54 | 0.09 | 33.95 | 2.79 | 14.95 | 0.92 | 0.55 | 0.36 | 62 | 299 | 5,832 |
| | Inferred | 20.63 | 0.23 | 0.09 | 30.26 | 5.32 | 14.34 | 1.21 | 0.60 | 0.61 | 172 | 297 | 3,798 |
| | Total | 183.34 | 0.50 | 0.09 | 33.53 | 3.07 | 14.88 | 0.95 | 0.55 | 0.39 | 75 | 299 | 5,603 |

**Table 14-30: Syerston scandium Mineral Resource by domain and classification category (300 ppm Sc cut-off)**

| | | Mt | Sc ppm | Sc metal | Sc2O3 |
|---|---|---|---|---|---|
| Alluvial | Measured | | | | |
| | Indicated | 1.12 | 368 | 411 | 629 |
| | Inferred | 7.29 | 366 | 2,671 | 4,086 |
| Overburden | Measured | 0.01 | 348 | 2 | 4 |
| | Indicated | 1.29 | 395 | 511 | 781 |
| | Inferred | 17.01 | 421 | 7,158 | 10,952 |
| Transition | Measured | 0.40 | 434 | 174 | 266 |
| | Indicated | 2.28 | 414 | 945 | 1,446 |
| | Inferred | 1.87 | 446 | 833 | 1,274 |
| Goethite | Measured | 0.91 | 512 | 467 | 714 |
| | Indicated | 2.82 | 443 | 1,251 | 1,914 |
| | Inferred | 3.98 | 536 | 2,133 | 3,263 |
| Sliceous Goethite | Measured | 0.44 | 439 | 191 | 293 |
| | Indicated | 3.05 | 401 | 1,223 | 1,871 |
| | Inferred | 3.19 | 392 | 1,252 | 1,916 |
| Total | Measured+Indicated | 12.32 | 420 | 5,175 | 7,918 |
| | Inferred | 33.34 | 421 | 14,047 | 21,492 |
| | Total | 45.66 | 421 | 19,222 | 29,409 |

**Table 14-31: Syerston scandium Mineral Resource by domain and classification category (No Sc cut-off)**

| | | Mt | Sc ppm | Sc metal | Sc2O3 |
|---|---|---|---|---|---|
| Alluvial | Measured | 23.51 | 47 | 1,099 | 1,681 |
| | Indicated | 40.24 | 94 | 3,763 | 5,758 |
| | Inferred | 59.39 | 128 | 7,618 | 11,656 |
| Overburden | Measured | 4.81 | 71 | 342 | 524 |
| | Indicated | 14.14 | 135 | 1,905 | 2,914 |
| | Inferred | 37.44 | 257 | 9,622 | 14,722 |
| Transition | Measured | 10.46 | 96 | 1,005 | 1,538 |
| | Indicated | 16.97 | 131 | 2,231 | 3,414 |
| | Inferred | 3.81 | 281 | 1,070 | 1,637 |
| Goethite | Measured | 32.24 | 70 | 2,254 | 3,448 |
| | Indicated | 36.53 | 92 | 3,343 | 5,115 |
| | Inferred | 8.09 | 336 | 2,716 | 4,156 |
| Sliceous Goethite | Measured | 26.11 | 38 | 979 | 1,498 |
| | Indicated | 40.39 | 62 | 2,521 | 3,858 |
| | Inferred | 8.72 | 234 | 2,043 | 3,126 |
| Total | Measured+Indicated | 245.41 | 79 | 5,175 | 7,918 |
| | Inferred | 117.45 | 196 | 23,069 | 35,295 |
| | Total | 362.86 | 117 | 28,244 | 43,213 |

### 14.2.23 Comparison with previous estimates

The cobalt grade of the Mineral Resource has increased by 30%. The Mineral Resource is now 101 million tonnes at 0.13% Co for contained cobalt metal of 132,000 tonnes. The nickel grade of the resource is 0.59% Ni for 593,000t of contained nickel. Of this total resource, 86% is in the Measured and Indicated categories.

This compares to the previously reported Mineral Resource (20 September 2016) of 109 Mt @ 0.10% Co and 0.65% Ni for 114,000t of contained cobalt and 700,000t of contained nickel.

Differences are due primarily to the re-interpretation of the laterite sub zones and changes in estimation procedures including unfolding methodologies, and high grade scandium and platinum modelling constraints.

The platinum in the Mineral Resource for the Project has increased significantly to 103 Mt @ 0.32 g/t Pt for 1,076,170 ounces, using a 0.15 g/t cut-off. Of this total resource, 94% is in the Measured and Indicated categories.

This compares to the previously reported Mineral Resource (20 September 2015) of 109 Mt @ 0.20 g/t for 700,888 ounces of contained platinum.

The scandium Mineral Resource for the Project has increased significantly to 45.7 Mt @ 420 ppm Sc for 19,222 tonnes of contained metal using a 300ppm cut-off. Of this total resource, 27% is in the Measured and Indicated categories.

This compares to the previously reported scandium Mineral Resource (17 March 2015) of 28.2 Mt @ 419 ppm Sc for 11,819 contained metal tonnes, using a 300 ppm Sc cut-off (i.e. an increase in contained scandium metal of 63%).

Key to this increase in scandium Mineral Resource was the detailed review of the deposit which established geological continuity of the scandium mineralisation and the definition of two distinct populations, specifically:

- A lower grade scandium resource overlying the main dunite basement, and included within the main zones of cobalt and nickel mineralisation, and
- A higher-grade scandium resource laterally surrounding the main zones of cobalt and nickel mineralisation.

### 14.2.24 Resource risk

Mineral resource estimates are not precise calculations, being dependent on the interpretation of limited information about the location, shape and continuity of the mineral occurrence and on the reliability of available sampling results.

The principal factors that may contribute to resource estimation error are:

Limited lateral continuity of unusually high cobalt grades. While there has been considerable lateral dispersion of low to moderate cobalt values, infill and twin drilling has showed that high cobalt values (above about 0.4% to 0.5%) can have very limited lateral continuity. Local infill drilling to 30 x 30 m in 2005 and definition of a high grade cobalt and scandium domains to limit extrapolation and smearing have ameliorated this risk, but it has not been eliminated completely.

Potential error in bulk density factors. These are based on limited numbers of direct measurements, which might not prove to be fully representative of the deposit as a whole.

# 15 Mineral Reserve Estimate

From the 2016 Mineral Resource, a mine plan was designed based only on Measured and Indicated Resource blocks. The pit designs are based on the 2005 open pit optimisation and subsequent design that utilised a 4.00 USD/lb, given the current Nickel price forecast is 7.50 USD/lb, the design presents a level of conservatism.

The Measured and Indicated Resources within the mine plan were converted to the Proven and Probable Mineral Reserves estimate presented in Table 15-1.

The calculated Mineral Reserve is based on an NSR value generated from the modifying factors below by applying the modifying factors discussed in the following section.

Financial viability of the mine plan was demonstrated at metal prices, modifying factors and parameters summarised in Table 15-2 and discussed further in Section 22.

The viability of the project is based on the metal price assumptions outlined in Section 19.3 (refer to Note 4 and 5 below).

SRK (Peter Fairfield) has completed a review of the relevant supporting information and examined relevant working files for validation and is satisfied that the validation checks and conclusions are correct as described. As the Qualified Person, Peter Fairfield takes responsibility for reported Mineral Reserve of this report.

**Table 15-1: Mineral Reserves**

| Category | Inventory (Mt) | NiEq (%) | Grade (% Ni) | Grade (% Co) |
|---|---|---|---|---|
| Proven | 59.48 | 0.96 | 0.71 | 0.10 |
| Probable | 44.23 | 0.83 | 0.58 | 0.10 |
| **Proven + Probable** | **103.71** | **0.91** | **0.65** | **0.10** |

Notes:

1) Tonnes are rounded to the nearest thousand.
2) Totals may appear different from the sum of their components due to rounding.
3) A cut-off grade based on NSR was used of approximately 0,25% NiEq, NiEq% = Ni% + (Co% * 2.95).
4) USD 7.50/lb nickel and USD12/lb cobalt and nickel recovery of 90% and cobalt recovery of 88.9%, USD:AUD of 0.75.
5) For economic modelling, a cobalt price of USD14/lb was used and USD7.50/lb for nickel.
6) The Mineral Reserve is a subset of the Measured and Indicated only schedule of a Life of Mine Plan that includes mining of Measured, Indicated and Inferred Resources.
7) The Mineral Reserve estimate was independently verified by Peter Fairfield, FAusIMM, CP (Mining), who is a full-time employee of SRK Consulting and a Qualified Person under NI 43-101.

**Table 15-2: Economic assumptions**

| Parameter | Unit | Rate |
|---|---|---|
| **Metal Prices** | | |
| Nickel price | USD/lb | 7.50 |
| Cobalt price | USD/lb | 14.00 |
| Exchange rate | USD: AUD | 0.75 |
| **Operating Costs** | | |
| Processing | | |
| Fixed G&A cost | AUD per annum | 10,089,471 |
| Variable processing cost | AUD/t ore | 26.20 |
| Acid cost | AUD/t | 7.00 |
| Beneficiation | AUD/t reject material | 2.00 |

| Parameter | Unit | Rate |
|---|---|---|
| Selling | | |
| Transport | AUD/t wet product | 5.00 |
| NSW Royalty | % | 4.00 |
| Ivanhoe Royalty | % | 2.50 |
| **Parameters** | | |
| Production rate | Mtpa plant feed | 2.5 |
| NiEq | % | Ni% + (Co% * 2.95) |
| Transition | | |
| Ni | % | 90.70 |
| Co | % | 90.80 |
| Goethite | | |
| Ni | % | 91.60 |
| Co | % | 89.90 |
| Silicified Goethite | | |
| Ni | **%** | **92.60** |
| Co | **%** | **90.80** |

Other material types assumed to have Goethite recoveries

## 15.1 Modifying factors

### 15.1.1 Mining dilution and recovery

#### Mining dilution

A review of the Resource model, which is based on 20 x 20 x 2 m blocks, and the mining will be carried out on 2 m high benches, it could be assumed that dilution was included within the Resource model.

#### Mining recovery

No mining losses were applied to the Mineral Reserves, for the following reasons:

- The deposit shows good lateral ore continuity
- There is broad width to the ore zones on individual benches
- Vertical and lateral ore contacts are likely to be gradational rather than sharp
- A detailed grade control program has been proposed
- Mining will take place on two-metre-high benches allowing for adequate ore definition.

### 15.1.2 Cut-off grade

Autoclave acid consumption, post beneficiation autoclave feed grades, and costs were applied to the blocks in the model to determine NSR and then reported by cut-off grade. The NSR in the mining model was calculated by incorporating the estimated processing cost (fixed and variable), metal recoveries, metal prices and the average acid consumption cost for different rock types. The NSR was calculated as the revenue less operating costs (excluding mining). The metal prices used for the block value were nickel USD7.50/lb and cobalt USD12.00/lb. Scandium has not included.

The Mineral Reserve is based on the marginal cut-off grade with a Block Value of AUD0.00/t which is approximately NSR AUD50.00/t or 0.25% NiEq. Each block was assigned a Low Grade (low grade), Medium Grade (medium grade) or High Grade (high grade) subcategory value for the purpose of maximising NPV in scheduling.

The cut-off grades used are listed below:

- Low grade: Value>0 $/t and NiEq<0.8%
- Medium grade: NiEq≥0.8% and NiEq<1.1%
- High grade: NiEq≥1.1%.

The primary parameters for the cut-off are USD7.50/lb Ni and USD12.00/lb Co, recoveries dependent on material type (Table 15-3), acid cost AUD77/t, fixed processing costs of AUD30/t, and NSW State royalty and a 2.5% Royalty to Ivanhoe Mines Ltd.

Mineral Reserves have been estimated by ore type and are summarised in Table 15-3.

**Table 15-3: Mineral Reserves by material type**

| Material | In situ Inventory (Mt) | Grade (% NiEq) | Grade (% Ni) | Grade (% Co) |
|---|---|---|---|---|
| Alluvium | 1.37 | 0.53 | 0.34 | 0.06 |
| Overburden | 0.54 | 0.46 | 0.28 | 0.06 |
| Goethite | 37.57 | 1.14 | 0.71 | 0.15 |
| Silicified Goethite 1 | 3.26 | 1.12 | 0.78 | 0.12 |
| Silicified Goethite 2 | 19.90 | 0.96 | 0.80 | 0.10 |
| Silicified Goethite 3 | 14.56 | 0.69 | 0.64 | 0.08 |
| Transition | 24.01 | 0.63 | 0.49 | 0.05 |
| Saprolite | 2.51 | 0.55 | 0.47 | 0.03 |
| **Total** | **103.71** | **0.91** | **0.65** | **0.10** |

Note: NiEq% = Ni% + (Co% * 2.95).

# 16 Mining

## 16.1 Mining methodology

All mining will be undertaken by conventional open pit methods, utilising backhoe excavators in the 100 t range, coupled with standard mine trucks. The actual equipment to be used will be left to the discretion of the selected contractor, allowing for the need to maintain ore selectivity within the pit and the high manoeuvrability requirement between the pits.

Mining activities will begin with the development of roads to permit access to all areas of the deposit and material haulage in all weather conditions. This will entail the provision of a hard and suitably drained road base. Establishment of all-weather access roads will be followed by carefully managed clearing of the mine sites themselves. It is proposed that the removal of vegetation will be carried out mechanically, followed by transport to a designated area.

Topsoil will then be removed and deposited in topsoil dumps. The method of removal is left to the discretion of the contractor, but most likely will involve the use of dozers to push the material into windrows for loading into the trucks.

Working benches will be 2 m high, for grade and selectivity considerations. The need for grade control and selectivity within the ore boundaries remains paramount for cash flow purposes. Where only waste is being mined, and depending upon local conditions, consideration will be given to operating with higher bench heights.

## 16.2 Mine design

Pit designs were not prepared for the 2016 PFS that forms the basis of this report. The pit designs from the 2005 Study were used to report from the 2016 mining model. Waste dumps and stockpiles were designed to accommodate the volumes in the mine schedule. The staging of the open pit mining operations is presented in Figure 16-1 to Figure 16-5.

In the 2005 Study, pit optimisations were undertaken on the 2005 resource model with relevant dilution, cost, revenue and geotechnical inputs taken into consideration and the optimisation pit shells were used for detailed pit design taking into account ramps and geotechnical considerations. The results of an updated pit optimisation are likely to be similar to the 2005 Study results.

SRK notes that the 2005 optimisations were undertaken using a nickel price of 4.00 USD/lb and the nickel price at the time of undertaking the feasibility study was 7.50 USD/lb.

It is recommended that updated pit optimisations and pit designs be undertaken in the further study work to confirm the pit staging and optimise the Mineral Reserve, ore and waste schedules.

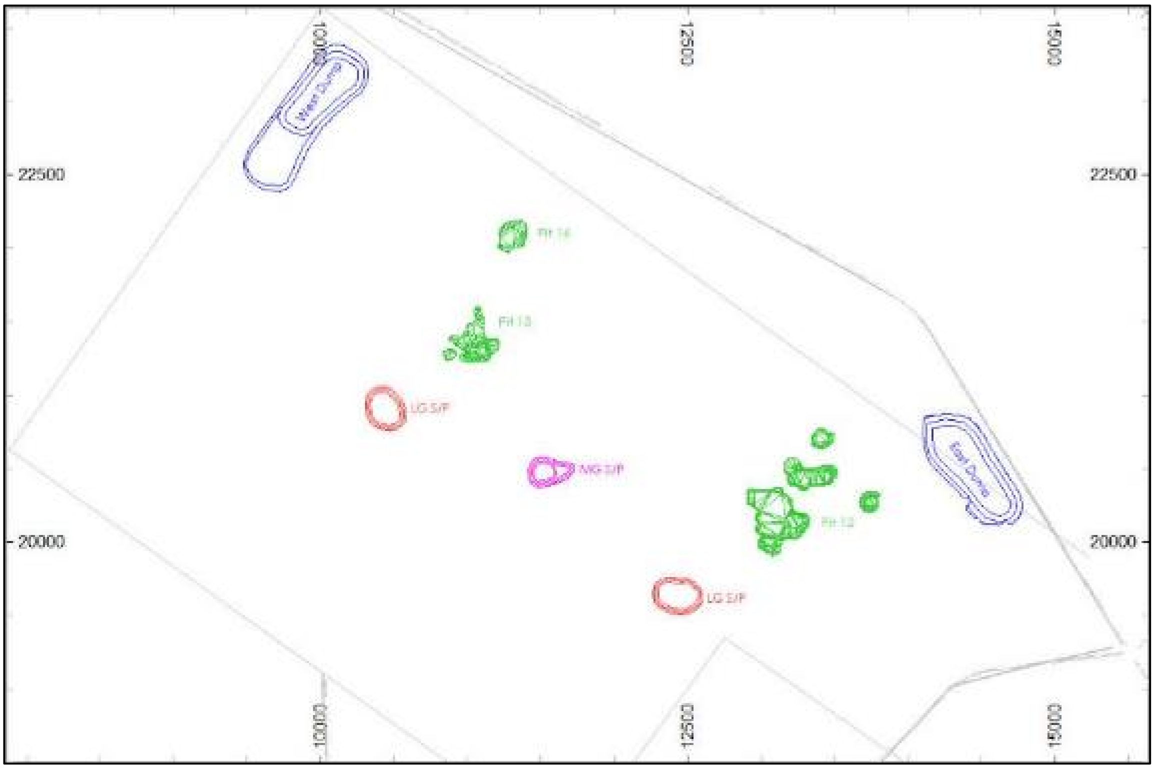



**Figure 16-1: Pit, stockpile and waste dump design – Year 1**

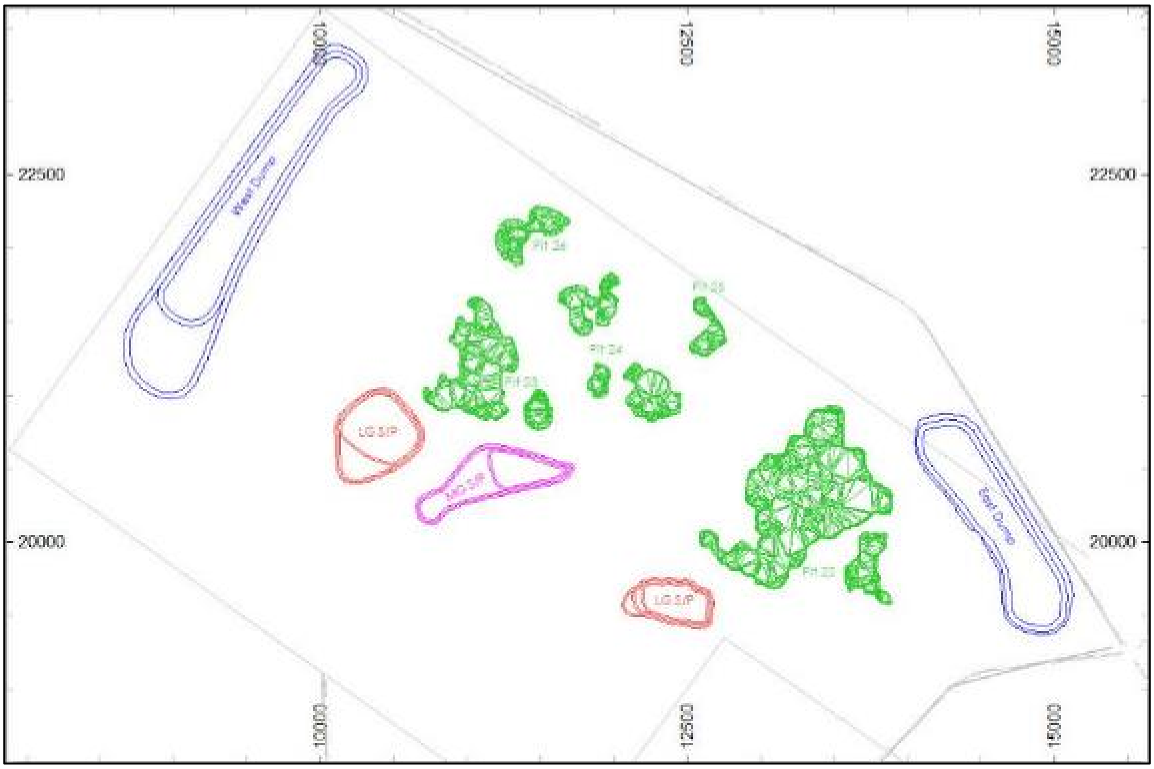



**Figure 16-2: Pit, stockpile and waste dump design – Year 5**


**Figure 16-3: Pit, stockpile and waste dump design – Year 10**

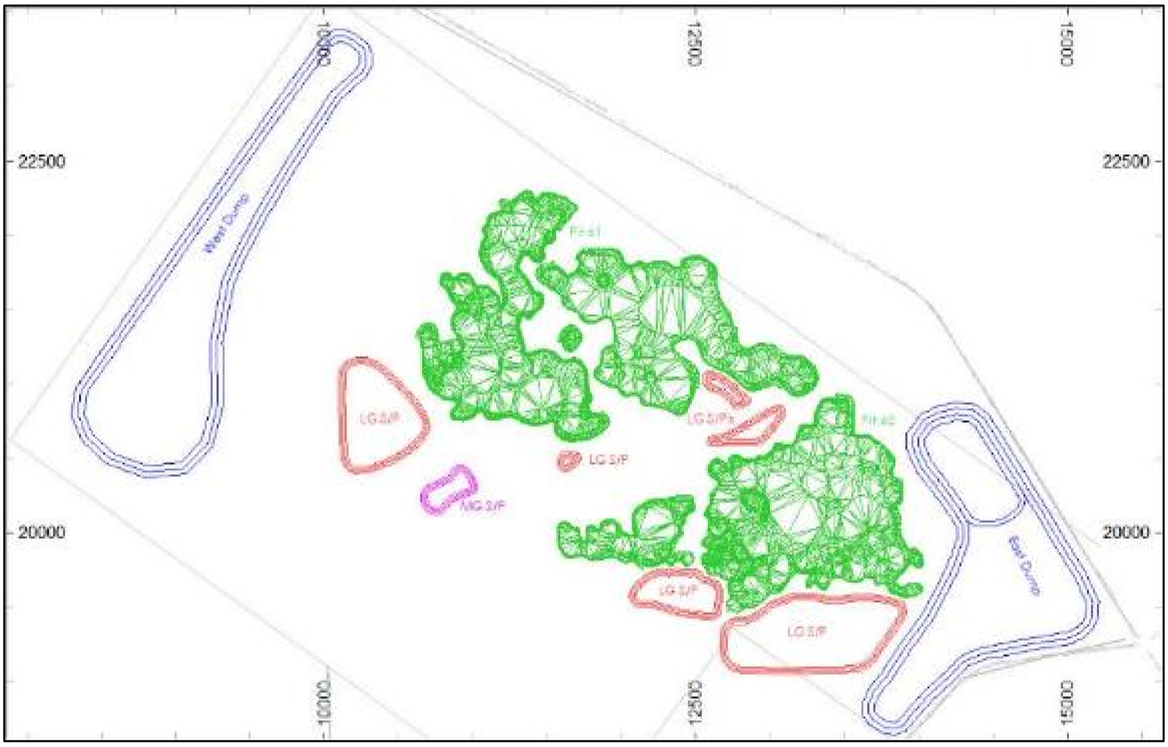



**Figure 16-4: Pit, stockpile and waste dump design – Year 20**

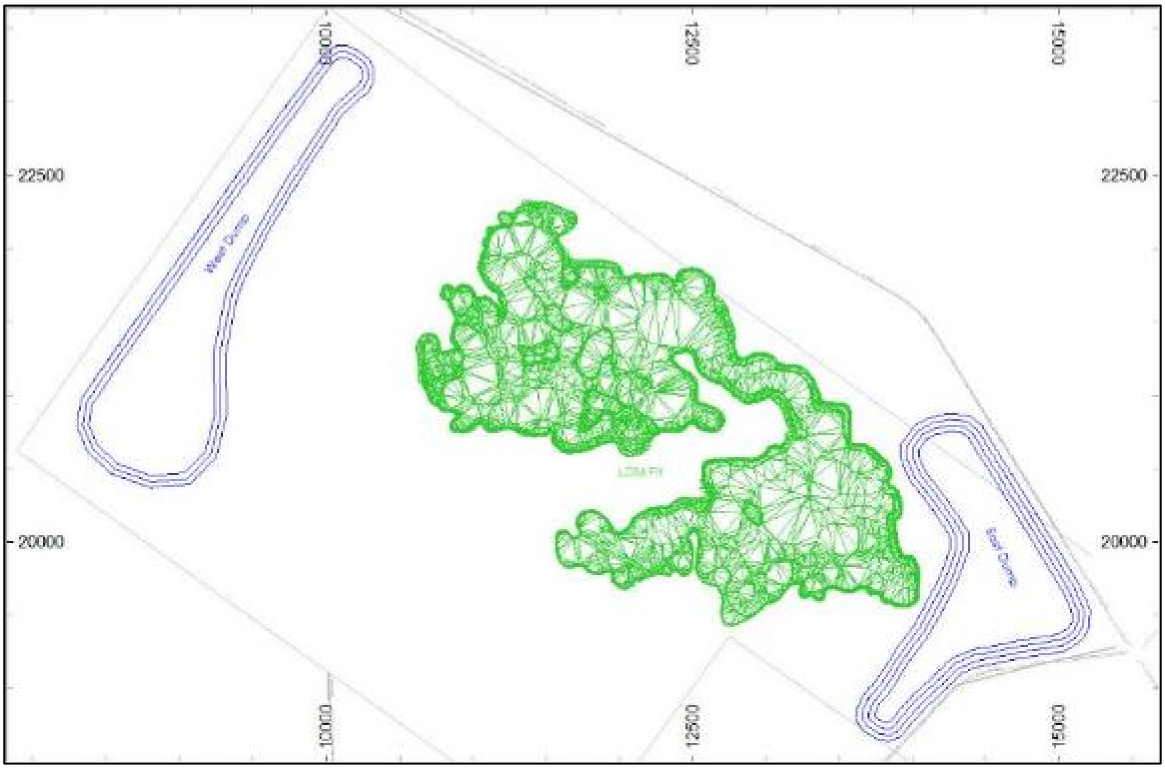



**Figure 16-5: Pit, stockpile and waste dump design – LOM**

### 16.2.1 Pit bench design

In June 2005 Golder carried out a review of its previous geotechnical assessment and concluded that the findings of its previous report (Geotechnical Investigation for the Plant Site and Limestone Quarry Syerston Nickel Project Fifield, NSW – March 2000) were still valid. The following section summarises the relevant part of the Golder report.

Slope stability was assessed using two-dimensional limit equilibrium stability analyses, assuming a 30 m deep pit, with a smooth overall slope (i.e. individual berms and batters were not modelled) inclined at 75° and 60°. For the analyses, Golder assumed that the overall slope failure would occur by shearing through the rock mass rather than sliding or opening along the discontinuities. This was mainly due to a lack of discontinuity data. The analyses indicated that a factor of safety in excess of 1.5 can be obtained for a 30 m high slope at 60°, using the lowest applicable strength parameters. However, some failure can still be expected due to the effect of discontinuities in the rock mass.

Golder noted that most of the steep walls of the old magnesite pits on the eastern margin of the deposit have stood-up well after some 15 years of inactivity.

It was anticipated by Golder that 5 m high batters will stand near vertical except those formed in residual soil, portions of the rock mass with adversely orientated discontinuities, zones of non-cohesive material and areas of excessive erosion due to rainfall and surface run-off.

Golder determined the design parameters to be those indicated in Table 5.1.1. The minimum berm width of 3 m was based on the use of trimmed and cleaned batters, whilst the batter angle should not exceed 75° to reduce the frequency and size of discontinuity induced batter failures. The pit designs have been completed using 5 m high benches, with 70° batters and 3 m berms every 10 m.

SRK reviewed the findings are considers them to be supportive of a PFS and the reported Mineral Reserve.

**Table 16-1: Slope design recommendations**

| Stratigraphic Unit | Minimum berm width (m) | Maximum batter height (m) | Maximum batter angle (°) | Inter-ramp slope angle (˚) |
|---|---|---|---|---|
| Residual soil | 3.0 | 5.0 | 45 | 0 |
| Laterites and siltstones | 3.0 | 10 | 75 | 30 |

## 16.3 Mining Schedule

OreWin used Datamine to report quantities and grades, and custom-built Excel spreadsheets for production scheduling, presented in Appendix A. The following steps were undertaken in the scheduling process:

- Definition of ore and waste within the pit limits using Datamine
- Production of bench reserves using Datamine
- Transfer of bench reserves to spreadsheet
- Produce preliminary schedule
- Review schedule with respect to mining practicalities and production improvements.

The methodology used for optimisation and design identifies designs that became the scheduling stages. Prioritising mining by the value of each stage and/or bench exposed, provides a schedule that both maximises value and is transparent when identifying mining positions in time.

Scheduling constraints and plant ramp-up schedule is proposed based are summarised as follows:

- Year One – 1.09 Mtpa feed
- Year Two – 2.11 Mtpa feed
- Year Three onwards – 2.5 Mtpa
- No Siliceous Goethite in Year 1
- Low grade plant feed for the first three months
- 50% low grade, 50% high grade plant feed in months 4 and 4
- High grade ore plant feed from Month 6 onwards
- From year one onwards, plant feed can include any material type.

Benches were scheduled in such a way as to keep metal production as smooth as possible, but with the ultimate aim of bringing metal forward. In some instances, more high grade material was mined than was required by the plant, the reason for this being that there was higher grade material on benches below that would increase overall metal production by accelerating mining.

After the initial plant start-up, all of the medium grade and low grade material will be stockpiled until the majority of the high-grade material is exhausted and high grade stockpile depleted.

To achieve the required high grade feed, as well as maintaining a reasonable mining profile, two large pre-strip periods were required. The first pre-strip period is over Years 1 and 2 (15 Mtpa), after which the mining rate is maintained for three years at approximately 10.0 Mtpa, until another ramp-up mining period is required over Years 6 and 7 (14.7 Mtpa). At this point the majority of the high-grade material has been mined, and the mining rate drops to approximately 5.0 Mtpa, and stockpile reclamation begins.

Mining for the first five years assumes three 100 t excavators and a fleet of twelve 90 t trucks. The flexibility in using a mining contractor allows the ability to add or remove machinery as required, especially over the two high pre-stripping periods. Wheel loaders, dozers and additional trucks will service stockpile rehandle as required.

### 16.3.1 Pre-production stripping and initial mine development

Site earthworks will be required for construction of the evaporation surge dam, the tailings dam, the evaporation ponds and the front end of the ROM pad and the limestone pad. This will require approximately 2.6 Mt of waste material.

Construction of the ROM and limestone pads, which will require approximately 1.3 Mt of waste material, will use material mined during the four-month pre-strip period for this purpose.

### 16.3.2 Schedule summary

Figure 16-6 and Figure 16-7 show the total material moved, the schedule has not been smoothed, over the life of the project as well as the stockpile profile.

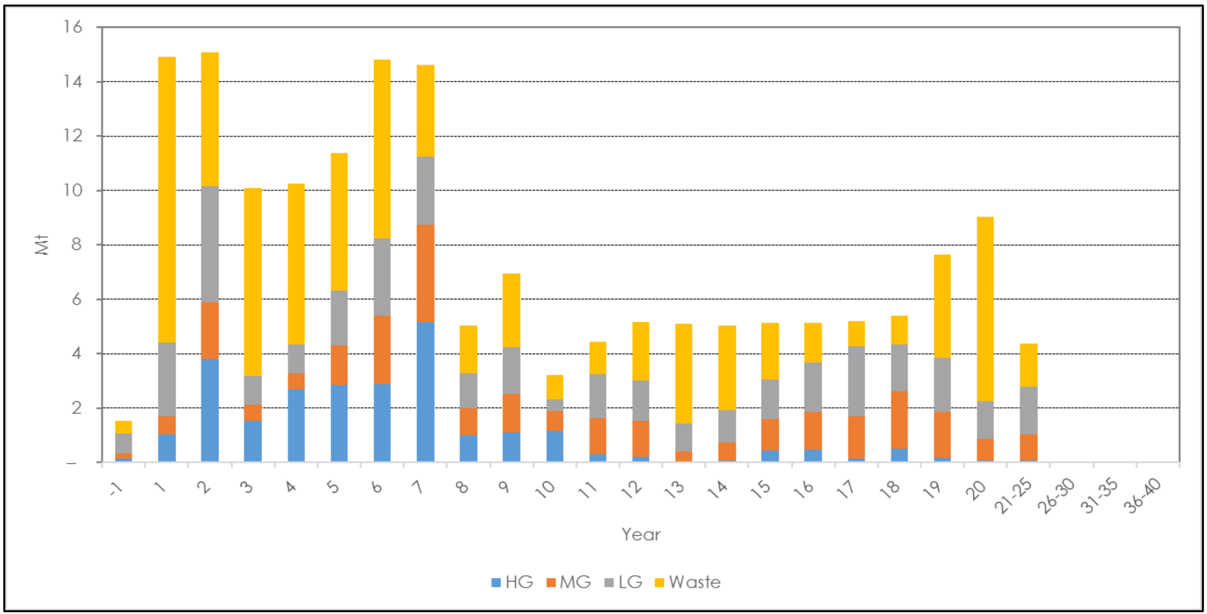



**Figure 16-6: Annualised mining schedule for LOM**

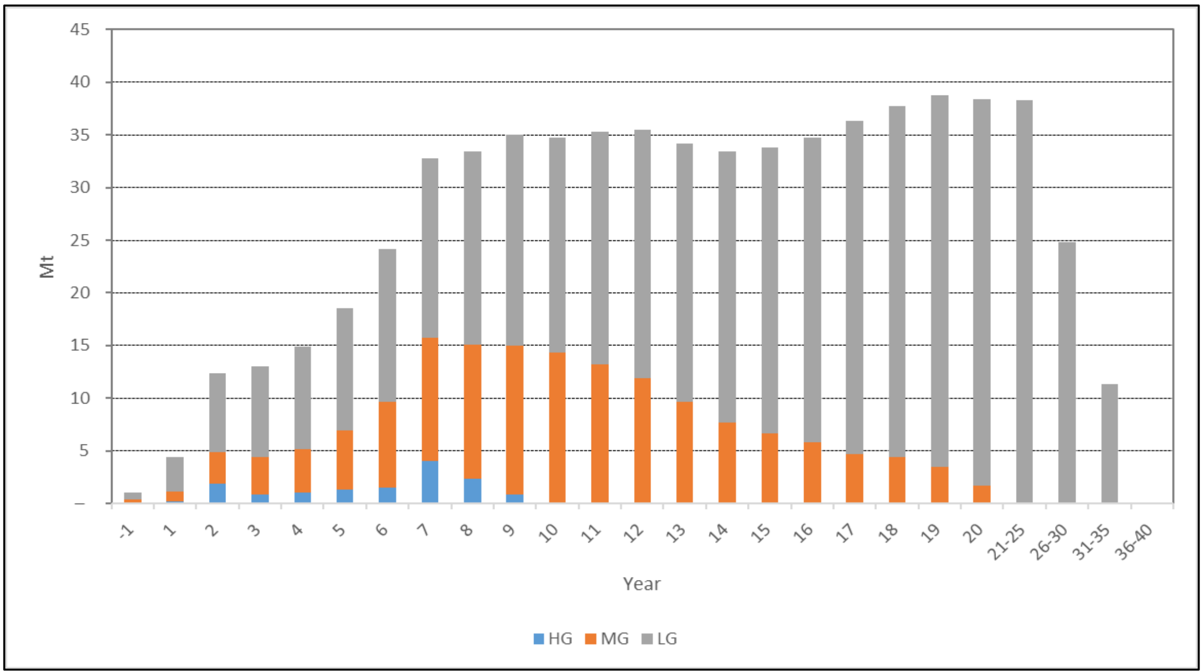



**Figure 16-7: Stockpile profile – LOM**

Figure 16-8 outlines the annualised plant feed by material type and Figure 16-9 shows the annualised contained nickel and cobalt contained in the feed. A breakdown is presented in tabular form, that was used in the techno-economic model in Appendix A.

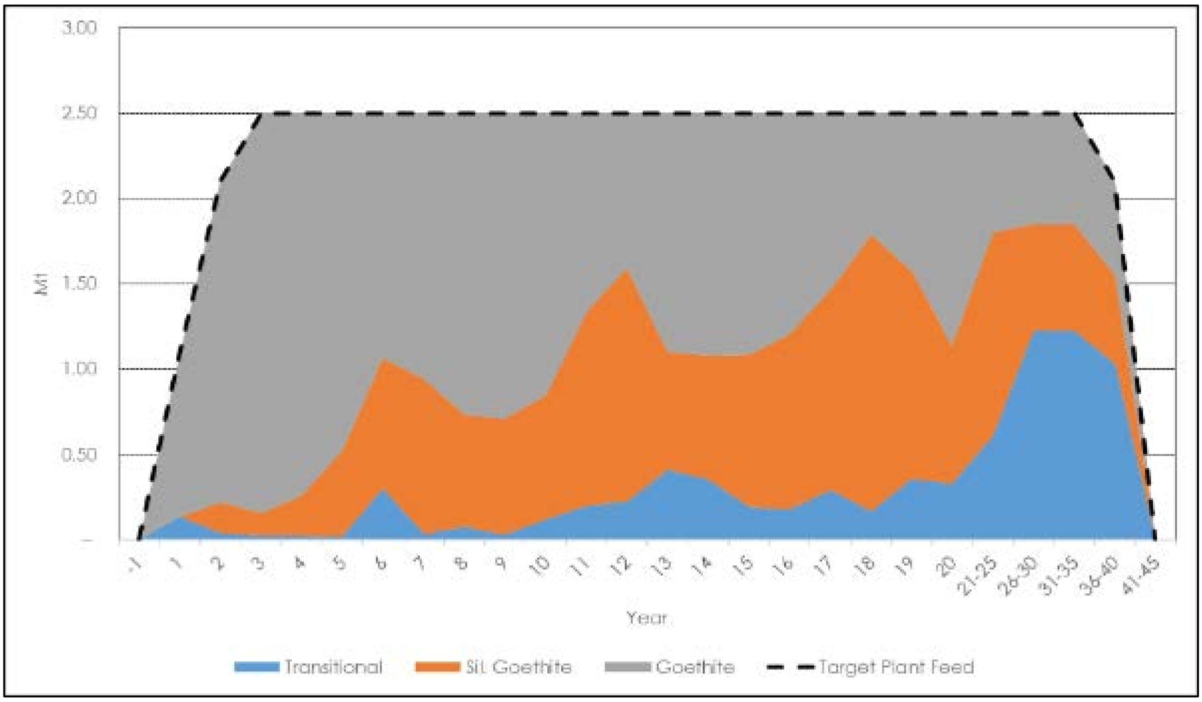



**Figure 16-8: Annualised plant feed by material type**

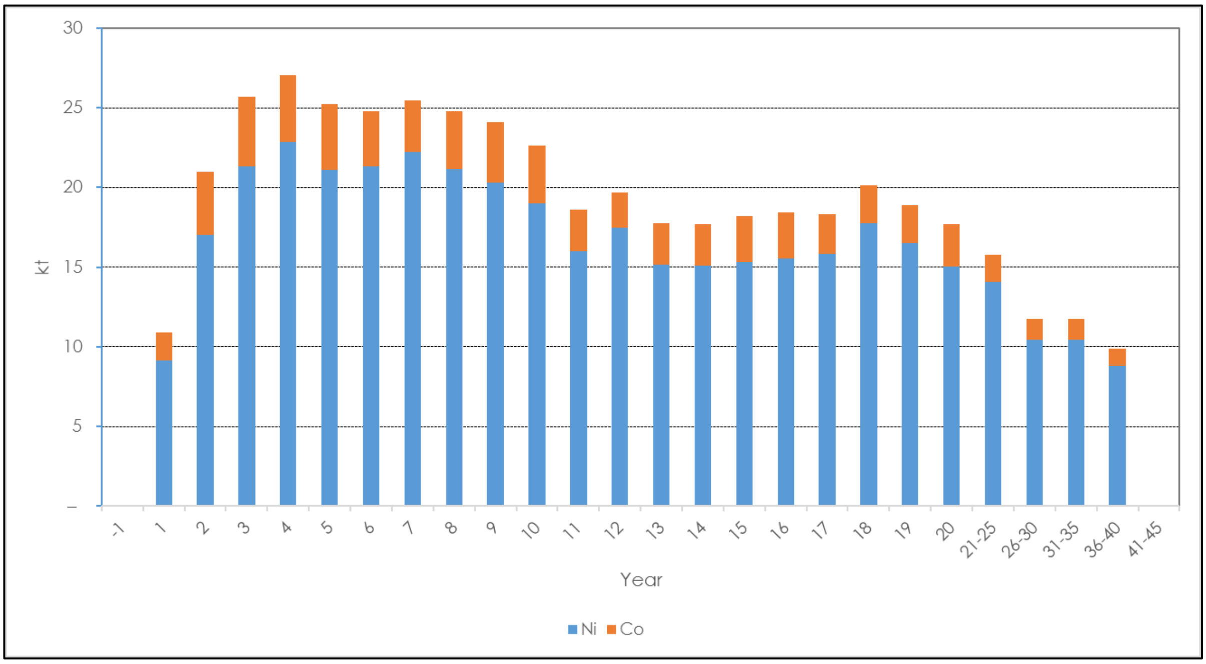



**Figure 16-9: Annualised contained metal in autoclave feed**

## 16.4 Mineability

Ore and waste dig rates of 440 - 490 bcm/hr have been assumed for the free dig material at Syerston. The mineability of the material depends predominantly on the depth of the pits. It does, in part, also depend on the equipment being used. Based on the reported on-site drilling performance, (PFS report), it has been inferred that the deposit could be free dug by powerful excavators.

## 16.5 Haulage

The trucks will convey the ore to either the ROM pad at the plant site or the stockpiles. A wheeled front-end loader will operate on the ROM pad. Some difficulties may be encountered in the overburden and orebody due to rainfall and surface run-off or perched water tables. The use of a suitable sheeting material on the roadways will alleviate these problems.

## 16.6 Contract mining

All mining operations will be carried out by a suitably experienced open pit mining contractor. This contractor will also be responsible for the mining-related construction activities and maintenance during operations. The contractor will also supply its own lighting equipment. Advantages of contract mining include:

- A reduction in initial capital
- Increased operational flexibility and the ability to change (ramp up and ramp down)
- Access, if required, to specialised services and equipment
- Mining costs are a relatively small part of the overall operating cost
- Focus of client attention on plant and more important issues.

The Scandium21 technical team will provide schedules of production requirements to the mining contractor who will be expected to deliver the appropriate quantities of ore and waste to the desired locations.

Other owner involvement will include:

- Management of the operation
- Management of grade control
- Compliance with the statutory obligations
- Supply of survey services
- Provision of a monthly measurement of the material moved for payment purposes.

The PFS assumed that the contractor will operate 20 hours per day. A work day will consist of two 10 hour shifts with a half hour break at mid-shift time. The actual number of working days per week will be left at the discretion of the contractor, provided it does not compromise the required production rate.

An area to the north-west of the ROM pad has been allocated as the contractor’s laydown. This area is available for the contractor’s office, workshop, fuel storage and equipment storage. The contractor is expected to supply all its own maintenance equipment and facilities. Scheduled maintenance and repair of all equipment will occur in the mine contractor’s workshops. Power will be supplied to the contractor’s laydown area by Scandium21.

## 16.7 Grade control

A separate contractor will carry out grade control drilling. Grade control will be carried out from surface in each pit area before mining starts. It is proposed that a drilling pattern of 7 x 15 m will be used. An RC drill rig with a face sampling hammer will be employed. Average hole depth from the surface to the bottom of the ore zone is 35 m. Approximately 182,000 m of drilling will be required each year, which includes grade control drilling in waste. It has been assumed that 75% of the holes will be sampled and assayed on site. This drilling will be completed during Year 9 of the mining operation.

The geological staff will use the information from grade control drilling to update all in-pit mapping, delineate ore limits and for reconciliation work. This information will be used to upgrade and refine the production schedule and mine plans.

## 16.8 Waste dumps

Two main dumps have been designed, which are located to the west and east of the mining operations. Figure 16-10 shows the approximate location of these dumps in relation to the LOM pit. Waste dumps have been designed to a maximum height of 30 m with batter slopes of 1 in 4. This complies with the development consent and EIS requirements, and is considered appropriate, given the need to maintain dump stability, reduce erosion and make maximum use of the areas available for waste dumping. A swell factor of 1.3 has been applied to all dumps. During construction, drainage channels will be utilised to control run off from the dumps.

A 30 m tree buffer corridor will be established and maintained around the site boundary to limit visual disturbance from the surrounding road and properties. Table 16-2 details the dump capacities.


**Figure 16-10: Waste dump locations**

**Table 16-2: Waste dump capacities**

| Dump | Capacity (Mbcm) | Capacity (Mlcm) |
|---|---|---|
| East | 23.3 | 30.3 |
| West | 17.9 | 37.6 |
| Total | **41.3** | **67.9** |

Dumps have been designed with sufficient capacity for the LOM with additional capacity on the west dump if required. Table 16-3 details waste dump requirements over the LOM. It is envisaged that once enough areas are mined out, waste material will be back-dumped into the pits rather than on the waste dumps, thus reducing the mining cost.

Longer term planning beyond the study time frame has considered locating an additional tailings dam (beyond Year 20) within the mined-out pits. Additional long-term planning and approvals will be required to ensure that suitable in-pit areas are available after 20 years. This may require advancing the eastern pit to final limits ahead of the western pit, to ensure availability of back-dump areas.

**Table 16-3: Estimated waste dump requirements**

| Cumulative to End of Year | Waste (Mbcm) |
|---|---|
| 1 | 6.34 |
| 5 | 20.3 |
| 10 | 29.8 |
| 20 | 46.6 |
| LOM | 52.0 |

Dumps will be constructed from the outside to allow the faces adjacent to the road to be rehabilitated at an early stage. Dumps will be progressively rehabilitated as they reach final design.

The majority of the waste material produced in the initial months will be used in ROM pad construction. In addition, approximately 2.6 Mt of waste material will be stripped by the earthworks contractor for use for site construction, for example tailings dams, evaporation ponds and evaporation surge dams, prior to the commencement of operations.

### 16.8.1 Waste material quantities

Total estimated waste mined over the life of mine is 83.3 Mt (52.0 Mbcm). Figure 16-11 illustrates the waste movement by period over the first 20 years.

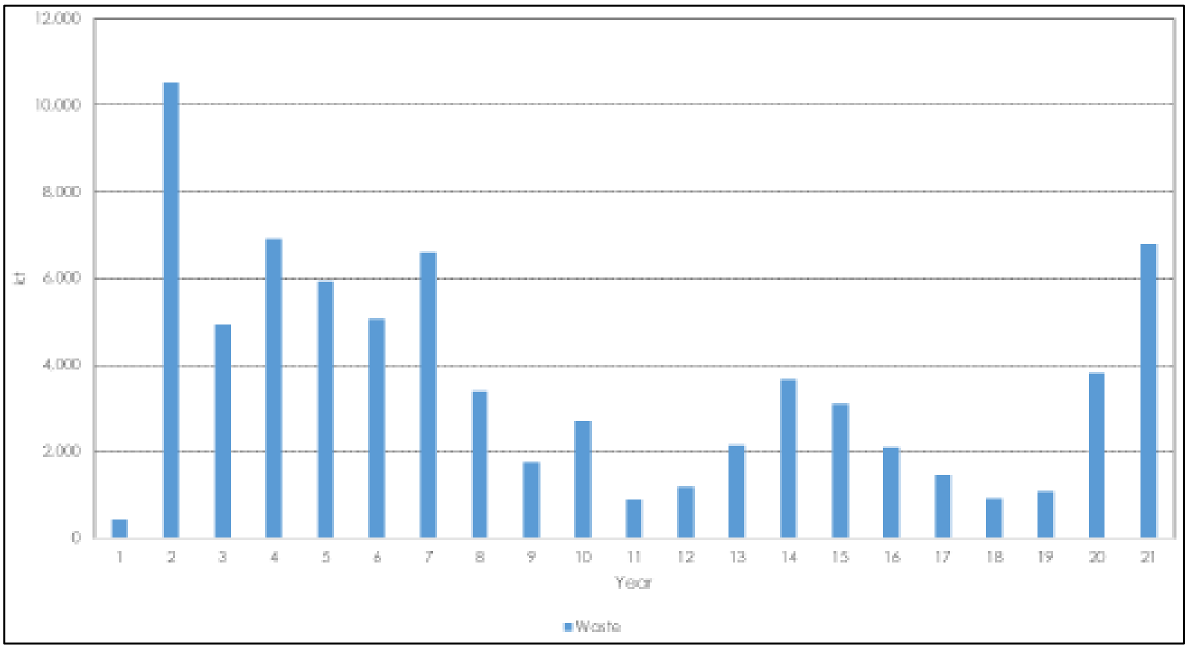



**Figure 16-11: Waste mining profile**

### 16.8.2 Topsoil

Topsoil to a depth of 150 mm is to be cleared and stockpiled from all intended plant locations, surge dams, evaporation ponds, tailings storage facilities, waste dump and stockpile areas. The only exclusion to this is in the area of the east dump, where extensive mining has already been carried out and there is minimal topsoil present. The mining contractor will be responsible for clearing all topsoil associated with the pits (apart from that removed by the earthworks contractor during construction), the dump areas and the stockpile areas, but excluding the material overlying the earthworks borrow pit area.

Topsoil dumps will be constructed to a maximum height of 6 m and in most cases, where the topsoil is unlikely to be required for a number of years, seeded with local grasses. This will have the effect of retarding soil sterility and staleness and reducing the effects of erosion.

### 16.8.3 Final reclamation

The Project has allowed for rehabilitation of waste dumps in line with the EIS. Dozers will produce a final slope profile in keeping with site limits and erosion control measures will be provided along with the relocation and spreading of the topsoil material. Flora and fauna will be established in line with the development consent and EIS requirements.

The parameters adopted for waste dump and stockpile design conform with Scandium21's desire to act in an environmentally responsible manner. Dump and stockpile design, construction and maintenance will be on going while the operation is in production. Areas available for rehabilitation will be rehabilitated as soon as practical, reducing disturbance and mining costs.

## 16.9 Westella limestone quarry

Limestone feed is required for acid neutralisation at the plant. There is a limestone deposit, Westella, located approximately 22 km to the south-east of the proposed Syerston Mine and Processing Facility, Figure 16-12.

The design work undertaken in the 2005 Study for mining limestone from Westella was used for the 2016 PFS, Figure 16-13. The operating costs were updated in the 2016 PFS.

High Grade limestone will be crushed and stockpiled at the quarry ready for transport by a separate contractor. Low Grade limestone and waste material will be stockpiled separately so that low grade can be reclaimed at a later stage, if required.

It was identified that there would be sufficient high grade limestone feed in the current model for approximately 17 years. Scandium21 indicated that the resource continues to the North, however further drilling is required to prove this. For the purpose of the mining study, it was assumed that the limestone resource continues to the North, and all mining constraints remained constant, i.e. SG, strip ratio, etc.

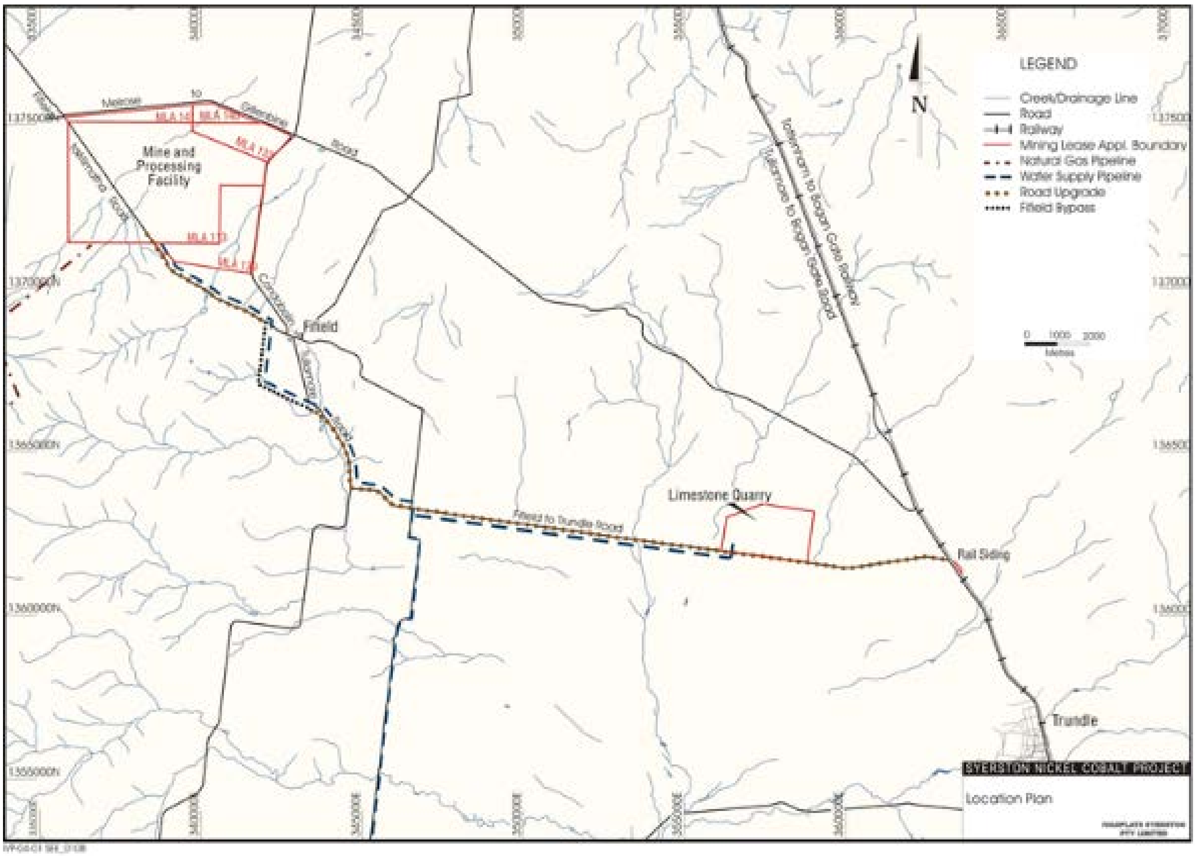



**Figure 16-12: Westella limestone quarry location**

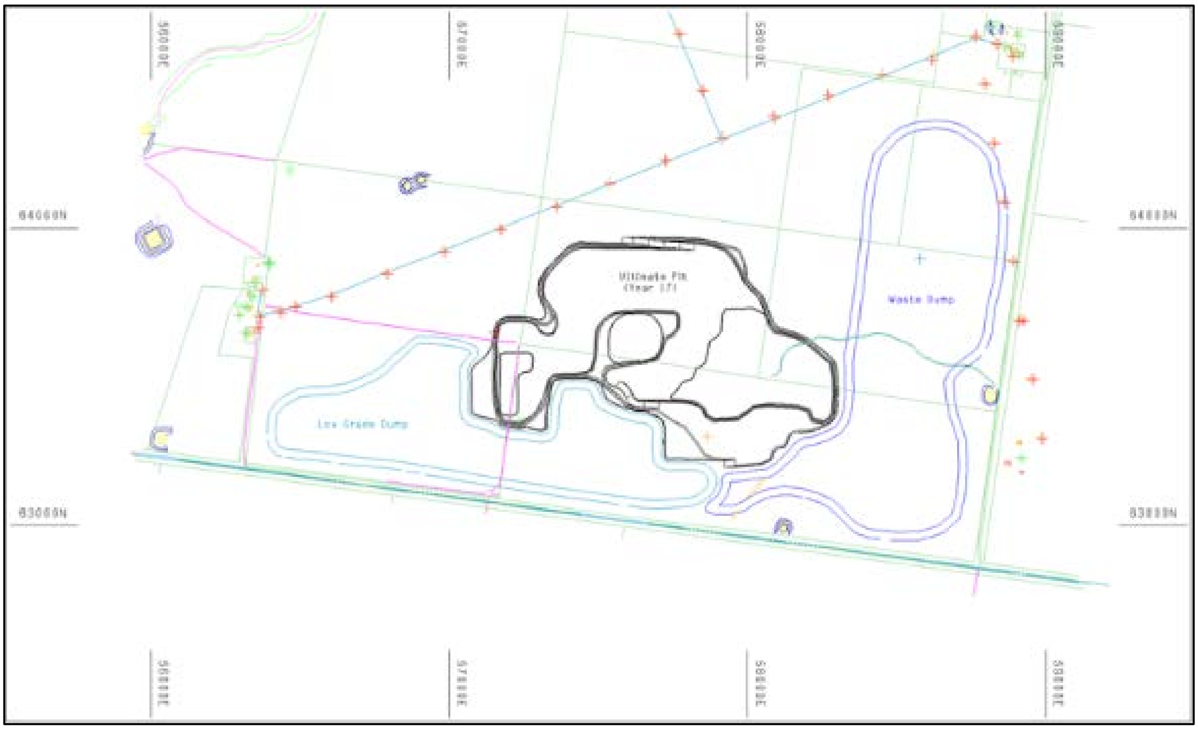


**Figure 16-13: Ultimate limestone design**

In the 2005 Study, pit optimisations were undertaken on the limestone resource model with relevant dilution, cost, revenue and geotechnical inputs taken into consideration and the optimisation pit shells were used for detailed pit design taking into account ramps and geotechnical considerations.

In 2005 a detailed schedule, including pit stage designs, was completed for the first five years of the quarry. As the deposit is fairly flat and regular, average stripping ratios (high grade, low grade an waste) were used to extend to the quarry schedule to the end of the mine life.

It is recommended that updated pit optimisation be undertaken in the further study work to confirm the pit staging, limestone and waste schedules.

The limestone quarry operations at Syerston will be carried out by a mining contractor. As a result, mining capital expenditure is reduced as the need to purchase major items of earthmoving plant and equipment is eliminated.

For the purpose of the Study it has been assumed that one mining contractor will mine both ore and limestone with two separate fleets, but maintaining one management. The costs sourced from budget contractor submissions provided by Scandium21 to OreWin are used as the basis for the pre-production capital costs.

For economic modelling purposes, all expenses incurred prior to any plant start-up are deemed to be pre-production capital. Pre-strip mining of the limestone deposit is started 1 month prior to the plant start up to create a stockpile of limestone at the mill.

Details of the operating costs are provided in Table 16-4.

**Table 16-4: Limestone mining operating costs**

| | Cost (AUD/bcm) |
|---|---|
| Load and Haul | 4.45 |
| Drill and Blast | 4.35 |
| HG Crushing | 4.73 |
| Limestone ROM Rehandle | 0.51 |
| Total | 14.03 |

# 17 Recovery Methods

## 17.1 Process development

The Syerston Ni/Co Project has been in development over an extended period with the original PFS completed in 1998 by Fluor Daniel Pty Ltd, followed up by a FS undertaken by SNC-Lavalin for Black Range Minerals in 2000 and an updated FS completed in 2005 for Ivanplats Syerston Pty Ltd by a SNC-Lavalin – JGC Corporation joint venture for a 2.5 Mtpa HPAL operation. Exploration began well before this, in the 1960s. The latest PFS completed in October 2016 leverages the previous detailed engineering deliverables.

Clean TeQ has been developing the nickel, cobalt and scandium RIP process since 2002 including three large scale piloting operations on laterite ore. Furthermore, Clean TeQ has developed a process for scandium recovery from titanium dioxide process streams and has operated a fully automated pilot plant operating on such process streams.

A separate FS for a scandium project was completed in August 2016. The development of the scandium project has been carried out to allow for the development of the separate and much larger nickel and cobalt operation which is Clean TeQ's development priority and the base case assumption of this PFS. As the high-grade scandium and nickel/cobalt zones are on separate areas these projects may be developed independently. In the case of joint development, the projects will be able to utilise common infrastructure and services, however it is the Company's view to develop only one Project at this time.

## 17.2 Syerston processing plant

The proposed Syerston Processing Plant is a complex hydrometallurgical processing flowsheet using conventional High-Pressure Acid Leach ("HPAL") to leach nickel and cobalt from the Syerston ores. The leached nickel and cobalt is then recovered through nickel and cobalt Resin-In-Pulp ("RIP") and solvent extraction before the final nickel sulphate and cobalt sulphate products are crystallised, dried, packaged and transported to market. Optionally a third scandium oxide product can either be produced from the HPAL discharge slurry (using RIP) or raffinate liquor streams (using IX), scandium refining and final calcination. The production of scandium is not incorporated in the base case. It remains as future Project upside.

The final slurry/solution after metal recovery is neutralised with limestone and sent to a tailings storage facility ("TSF") and an evaporation pond. The process plant will produce high purity hydrated nickel and cobalt sulphate products, as well as the option of a 99.9% $Sc_2O_3$ product.

Commercial HPAL processing of lateritic nickel and cobalt ores commenced at Moa Bay in the late 1950's. This was followed in the late 1990s with the construction of the Murrin Murrin, Cawse and Bulong operations. The technology is now into what is often referred to as the fourth generation of HPAL projects and operations with examples such as Ravensthorpe, Coral Bay, Ambatovy, Taganito and Goro. While some of these operations have experienced materials of construction, start-up and operational issues, the technology is considered to be well established.

Clean TeQ uses a proprietary ion exchange technology (Clean-iX®) for extraction and purification of metals from the liquor component of HPAL discharge slurries. The development of the base technology for the Clean-iX® process was developed by the All Russian Research Institute of Chemical Technology (ARRICT) over a period of 40 years. In 2000, Clean TeQ Ltd (obtained the exclusive licence for all technical information relating to ion exchange resin, ionic membranes, organic solvent extractants, including manufacturing know-how and plant design, for all countries outside the former USSR. Since obtaining the licence, Clean TeQ has further developed the technology for base metals, uranium and gold, with particular improvements for laterite ore processing, scandium and uranium. Clean TeQ has been granted 10 additional patents on various aspects of the technology, including one for extraction and purification of scandium.

Conventional flowsheets used in the industry today use counter current decantation (CCD's) followed by precipitation of either a mixed sulphide or hydroxide intermediate, re-leaching and solvent extraction (SX) to recover the metals from leached slurries. This process has several disadvantages including: higher capital and operating costs, as well as lower metal recoveries. The use of Resin-In-Pulp (RIP) technology addresses many of these issues. RIP uses solid ion exchange resin beads which are contacted directly with the leached slurry to extract more than 98% of the contained metal in the solution. Ion exchange resins are ideal for recovery and concentration of lower concentration metals, which is the case with lower grade laterite resources. This means that plant size and chemical costs are reduced compared to SX.

The summary flowsheet, including the option of scandium production, is shown below in Figure 18-2. Note that it shows the optional scandium oxide production option however it is not currently incorporated into the detailed flowsheet on which the capital and operating costs have been estimated.

The process can be broadly defined as:

- Ore Preparation and Milling
- HPAL
- Scandium RIP
- Scandium Purification (optional only – not base case)
- Nickel/Cobalt RIP
- Nickel/Cobalt Sulphate Purification and Recovery
- Tailings neutralisation and storage
- Process reagents and utilities (sulphuric acid, steam, water, limestone, other).

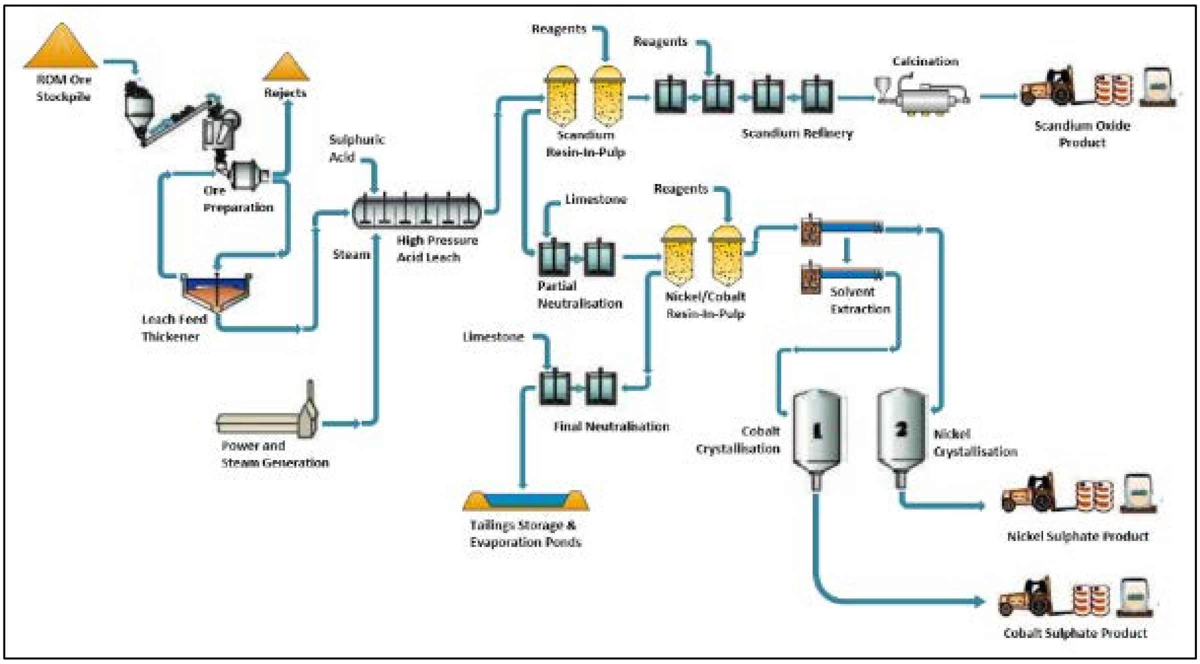



**Figure 17-1: Processing flowsheet summary**

Further descriptions of each of the key process are provided in the following section. Additional detail is provided in the PFS.

### 17.2.1 Key design criteria

Based on historical data taken from the 2005 Nickel/Cobalt Feasibility Study Update, the 2016 Scandium Feasibility Study and Clean TeQ's internal database of nickel and cobalt recovery using RIP, a Process Design Criteria was developed for the 2016 PFS.

The operation processes 2.5 Mtpa of feed with a plant availability of 89.6%. Nickel and cobalt leach extractions are above 95% and estimated overall recoveries are 93.5% and 92.7% respectively. The key criteria are shown in Table 17-1.

**Table 17-1: Process design criteria summary**

| | Units | Quantity |
|---|---|---|
| Autoclave Throughput | tpa | 2,500,000 |
| Operating Hours per year | hours p.a. | 7,690 |
| Availability Leach Plant | % | 89.6 |
| Mine Life | years | >20 |
| Average Product Production Post Ramp-Up (Year 3 - 20) | | |
| Nickel Sulphate ($NiSO_4.6H_2O$) | tpa | 85,136 |
| Cobalt Sulphate ($CoSO_4.7H_2O$) | tpa | 15,490 |
| Nickel Equivalent Production | tpa | 18,730 |
| Cobalt Equivalent Production | tpa | 3,222 |
| Scandium Oxide ($Sc_2O_3$) - Optional | tpa | ~ 50 |
| Nickel Grade (Average) | % | 0.80 |
| Cobalt Grade (Average) | % | 0.14 |
| Nickel Grade (Range) | % | 0.7 - 1.0 |
| Cobalt Grade (Range) | % | 0.1 – 0.2 |
| Scandium Head Grade | ppm | 40 – 60 |
| Autoclave Operating Temperature | ºC | 250 |
| Autoclave Residence Time | minutes | 70 |
| Sulphuric Acid to Leach | kg/t ore | 240 - 290 |
| Estimated Leach Extractions (Goethite) | | |
| Nickel | % | 97 |
| Cobalt | % | 95.5 |
| Scandium | % | 86 |
| RIP Metal Recovery (Ni/Co & Sc) | % | 98 |
| Estimated Overall Recoveries (Goethite) | | |
| Nickel | % | 93.5 |
| Cobalt | % | 92.7 |
| Scandium | % | 83.4 |

The mine production schedule is based on the 2005 Feasibility Study Update. In that study, the production of cobalt was capped at 5,000 tpa as the main focus was nickel production. However, there is potential to alter the mine plan such that higher grade cobalt is mined, particularly in the early years of operation. This is a current opportunity given the currently strong market for cobalt.

### 17.2.2 Ore preparation and milling

The primary design goal of the ore preparation circuit is to produce feed slurry thickened to an estimated 45-48% w/w solid. To achieve this density specification, the ore preparation plant requires feed that has been blended such that the slurry produced has rheological and settling properties that are amenable to thickening and pumping at this density.

Two (2) ore feed types are processed through the plant, a goethite and siliceous goethite feed. Both are prepared through the same crushing and milling circuit but utilise different operating regimes and screening equipment. Additional screening equipment is required for the beneficiation of siliceous goethite feed for the removal and rejection of coarse low-grade material. The two ore types are campaign treated through the circuit according to the timing for access to the ore types as dictated by the mine plan.

The circuit is conventional, comprising a run of mine (RoM) feed bin, apron feeder, mineral sizer style crusher, SAG milling and particle size classification through hydrocyclones. The slurry product density is increased to design through a Leach Feed Thickener, the underflow of the thickener is pumped to the Leach Feed Surge Tanks.

Consideration of the materials handling properties of the clay component of the ores is made through the selection of bin angles, equipment type, specifically the mineral sizer, the use of hot process water and other initiatives.

### 17.2.3 High pressure acid leaching

The circuit is designed to extract nickel, cobalt and scandium from the lateritic ores using high pressure, temperature and acid. The proposed HPAL circuit consists of two separate operating trains. Each train comprises a three-stage direct contact heater system, two stages of centrifugal pumping and a final stage of positive displacement pumping, a pressure leach autoclave, and three stages of flash tanks for pressure letdown and steam recovery.

Thickened slurry is heated to approximately 239°C in three stages using direct contact steam heaters. The first two heaters use steam recovered from the autoclave discharge flash tanks, while the final heater uses fresh 6,000 kPag high pressure (HP) steam from the steam header. The heated slurry is fed into a high-pressure autoclave with six agitated compartments. Sulphuric acid (98.5% concentration) is added into compartments one and two and due to the exothermic reaction, the target operating temperature of 250°C is obtained. Nickel, cobalt, scandium and other elements are leached from the ore into solution, and non-condensable gases are bled from the autoclave. The bulk of the iron reprecipitates (as a solid) before discharging the autoclave. The design of the titanium clad autoclaves considers sufficient mixing, residence (reaction) time, scaling and corrosion rates and materials of construction.

Slurry exits the last compartment of the autoclave through a controlled pressure let down circuit. The high autoclave operating pressure is let down in three stages of flash tanks. As the hot, pressurised slurry passes through the choke valve, it accelerates rapidly, releasing energy. The resulting pressure drop causes some of the water in the slurry to flash to steam. The steam released in the first two stages is used to preheat the incoming slurry while the steam from the final atmospheric flash stage is used to heat process water. The levels of these vessels are managed through modulating ceramic lined choke valves, with a fixed choke valve aperture.

A plant control system will ensure that the primary design goals are accomplished and that the equipment can be operated safely and easily.

The design parameters are to:

- Extract to solution the maximum amount of nickel, cobalt and scandium whilst minimising the co-extraction of impurity elements, particular iron, alumina and silica;
- Minimise the usage of steam and acid whilst still achieving optimal leach extractions;
- Recover flashed steam from the 1st and 2nd stage Flash Vessels to be used in the 1st and 2nd stage Direct Contact Heaters;
- To inject sulphuric acid into the first and second compartments of the autoclave;
- Addition of steam from the acid plant to the 3rd stage trim heater to maintain autoclave temperature at 250°C; and
- The ability to sparge trim steam directly into the autoclave to aid in start-up of the autoclave and to maintain operating temperature if the need arises.

This process description is for both trains of the HPAL Circuit; each train is identical and has the same type of equipment and configuration. Some supporting circuit equipment is shared by both trains, such as the HP seal water supply and vent systems. Most pumps also have a duty and standby arrangement in the HPAL area.

### 17.2.4 Scandium RIP and recovery (optional)

The scandium recovery circuit is optional and was not considered for the Syerston Nickel Cobalt PFS.

### 17.2.5 Nickel/ cobalt RIP

The nickel and cobalt contained in the HPAL discharge is recovered through a number of steps. A number of them are conventional in laterite processing, however the Resin-In-Pulp, whilst adopted by other metal processing flowsheets and long developed for nickel laterites to a pilot level of operation, is not yet fully commercialised.

The HPAL discharge slurry is first treated with hydrogen sulphide ($H_2S$) gas to reduce the chromium from its hexavalent (+6) form to its much less oxidising trivalent (+3) form. This protects the resin and solvent extractants downstream, while also precipitating the copper contaminant as copper sulphide (CuS).

The pre-reduced liquor is partially neutralised with limestone to a pH above 3.5 in a series of four tanks to improve the selectivity of the nickel and cobalt in RIP over other impurities such as iron and aluminium. Gypsum and other sulphates are precipitated in the process and the generated carbon dioxide gas is vented to atmosphere. Limestone is sourced from a local quarry.

Partially neutralised slurry is screened before contacting with resin to ensure clean separation of loaded resins from slurry. The nickel and cobalt contained in the liquor selectively recovered onto a resin in a 10-stage counter current RIP circuit using air agitated Pachuca tanks, with a recovery of over 98%. Loaded resin is screened and washed to remove solids, and then undergoes desorption using weak sulphuric acid generating a pregnant liquor (nickel and cobalt sulphate liquor). The barren resin is then washed with water to recover acid.

### 17.2.6 Nickel/ cobalt sulphate purification and recovery

The pregnant liquor from desorption is then purified and the final sulphate product recovered. The eluate stream is initially treated through an impurity removal solvent extraction stage to remove zinc, iron, aluminium and copper. The resulting treated liquor is sent to a small solvent extraction system whereby cobalt is loaded onto an organic, leaving the nickel in the raffinate. The nickel-bearing raffinate is fed to another nickel purification stage of solvent extraction step where it is extracted, scrubbed and stripped. The production is sent to a nickel sulphate triple affect crystalliser to produce high purity hydrated nickel sulphate ($NiSO_4.6H_2O$) products. The supernatant from the crystalliser is recycled back to the Desorption step, as it contains small amounts of nickel.

The cobalt-bearing organic is sent to a scrub stage to remove impurities and small amounts of nickel before being sent to the cobalt stripping step, whereby cobalt is stripped from the organic using dilute sulphuric acid. The cobalt-bearing strip liquor is sent to a cobalt sulphate crystalliser to produce high purity hydrated cobalt sulphate ($CoSO_4.7H_2O$) products. The supernatant from the crystalliser is recycled back to the cobalt stripping stage, with make-up sulphuric acid added.

The nickel and cobalt products are run through a rotary dryer, and stored in product bins in preparation for packaging. Both the nickel and cobalt sulphate products are packaged in 1 tonne lined bulk bags in separate bagging facilities before dispatch to market.

An ammonium sulphate (Amsul) by-product is made for the fertiliser market. Barren liquor is processed through a triple affect crystalliser, dried and stored in bulk product shed. From here it is loaded by front end loader into side or back tipping trucks and transported to market. The product a minimum nitrogen and sulphur concentration of 21% w/w and 24% w/w respectively.

### 17.2.7 Tailings neutralisation, storage and evaporation

Tailings are neutralised with limestone slurry and air addition to remove free acid and precipitate the metal ions as stable hydroxides prior to discharge to the tailings dam. The metals are thus captured in the solids, minimising any environmental impact through leaching from the tailings. This is undertaken in two agitated tanks. The discharging slurry is thickened in a Tailings Thickener to increase the slurry density before being pumped through a single pipeline to the Tailings Storage Facility (TSF) for final storage.

The proposed TSF is located approximately 500 metres from the processing plant. It is made up of two cells. Tailings discharge will be in one cell at a time, with the other cell undergoing wall lift construction. Tailings are distributed around each cell with spigot take-offs every 10 metres, ensuring even sub-aerial deposition. Each cell has a central decant tower with access causeway, for control of the supernatant water pond size. Toe Drain Pumps are also available to collect any excess, unexpected drainage from the TSFs. They will pump the liquid back into the TSF.

Decant water flows from the TSF, are recovered through the central decant towers, to a common decant dam from which it is pumped to a series of seven shallow Evaporation Ponds. An evaporation pond is incorporated into the design to reduce excess water requirements due to the positive water balance and the need to bleed liquor to maintain several elements below their solubility limit.

A Decant Solution Surge Pond takes overflow from the Decant Dam and Tailings Dams during high rainfall periods or as required. Excess solution from the Decant Solution Surge Pond is pumped by the Surge Pond Return Water Pond Pumps to the Decant Dam as available or for use as Mine Water for haulage road dust suppression.

### 17.2.8 Reagents and utilities

The Syerston Plant is a complex hydrometallurgical flowsheet with several reagents and utility requirements. Key reagents and utilities include:

- Sulphuric acid
- Limestone
- Sodium carbonate
- Water

- Natural gas/ power
- Steam.

Sulphuric acid is produced on site with a dedicated acid plant capable of making 2,700 t/d of sulphuric acid. This incorporates a sulphur receival system incorporating sulphur handling, stockpile, and reclaim areas designed to receive sulphur delivered by end tipping road trains at regular intervals. Sulphur would be railed to a local siding located nearby between the towns of Trundle and Fifield and loaded into end tipping road trains.

Sulphur is melted, filtered, then burnt in air at 1200ºC to produce sulphur dioxide ($SO_2$) gas. This is converted to $SO_3$ then absorbed in sulphuric acid with make-up water added to an absorber tank. Product acid (98.5%) is withdrawn from this circulating acid, stored and is distributed around the plant. The majority is used in the HPAL circuit. A road tanker unloading facility is provided so that sulphuric acid production can be supplemented by imported acid for plant start up and for periods if consumption is high.

The acid plant will also produce high pressure (HP) steam at 6,000 kPa and 450°C. This steam is exported to the Power Plant where it is desuperheated for distribution to the process plant users. Any surplus steam is used in the power plant for power generation.

Limestone is mined nearby by a contractor and delivered to the Limestone ROM stockpile crushed to 100% passing 200 mm. The limestone quarry is discussed in more detail in Section 18 Project Infrastructure. The Limestone plant produces limestone slurry at an estimated 40% solids, ground to a $P_{80}$ of 75µm using a simple impact crusher, overflowing ball mill and hydrocyclone classification system. The feasibility study will consider the use of three stages of crushing and product thickening based on the experience of crushing and grinding calcrete at Murrin Murrin. Product limestone slurry is contained in storage tanks and distributed to the plant via a ring main. The bulk of the limestone is consumed in the partial neutralisation and tailings neutralisation areas.

There are several other reagents required by the plant. These additional chemicals and general freight will be trucked or railed to site in either B-Doubles, ISO containers, IBC's or bulk bags and stored on site in a dedicated reagents area. Sodium carbonate will be sourced in bulk bags.

Extractants, sodium hydroxide, oxalic acid and ammonium hydroxide will be required in the purification circuit. These reagents will be sourced in IBCs and stored adjacent to the purification circuit. Reagents will be pumped to the purification circuit using chemical dosing pumps. Solvent extractant diluent will be delivered in tankers and stored in tanks on site.

Flocculant will be used in thickener operations in the leach feed thickener and tailings thickener. Provision for two powder flocculant make-up systems has been included.

Natural gas, steam, power and water are discussed in Section 18 (Project Infrastructure).

## 17.3 Products

### 17.3.1 Hydrated nickel and cobalt sulphates

The process plant has been designed to produce high purity hydrated nickel sulphate ($NiSO_4.6H_2O$) and hydrated cobalt sulphate ($CoSO_4.7H_2O$) products. The product from the elution circuit of the Resin-In-Pulp plant is a high concentration high purity combined nickel and cobalt sulphate solution. Therefore, the process is ideally suited to the battery sector, which requires sulphates for precursor production, and potentially eliminates process steps that exist in the current cathode supply chain. Key nickel and cobalt product specifications in the target grade and in the maximum grade products are shown below in Table 17-2.

**Table 17-2: Nickel & cobalt product specification**

| Product Specification | | Ni Product | | Co Product | |
|---|---|---|---|---|---|
| **Element** | **Unit** | **Target Specification** | **High Specification** | **Target Specification** | **High Specification** |
| Ni | % w/w | 22.1 | **22.3** | **< 85 (ppm)** | **8 (ppm)** |
| Co | % w/w | < 669 (ppm) | **5 (ppm)** | **20.5** | **21.0** |
| Ca | ppm w/w | 23 | **4** | **15** | **3** |
| Cl | ppm w/w | 49 | **2** | **65** | **20** |
| Na | ppm w/w | 55 | **10** | **20** | **10** |
| Mg | ppm w/w | 23 | **1** | **13** | **10** |
| $NO_3$ | ppm w/w | 10 | **10** | **233** | **100** |
| Ag, Al, As, Cd, Cr, Cu, Fe, Hg, K, Mn, Pb, Se, Si, Sn | ppm w/w | < 10 | **≤ 5** | **≤ 10** | **≤ 10** |
| Zn | ppm w/w | 71 | **1** | **7** | **3** |
| Water insoluble fraction | ppm w/w | 71 | **20** | **86** | **25** |
| Moisture | % w/w | < 0.5 | **< 0.5** | **< 0.5** | **< 0.5** |

### 17.3.2 Ammonium sulphate

An ammonium sulphate (Amsul) by-product is made, primarily targeting the fertiliser market. Amsul products have a mean product size above 1.5 mm and a target size range of 40% w/w between 1 and 3 mm. This is dried below 1% moisture and stored in bulk product shed. From here it is loaded by front end loader into side or back tipping trucks and transported to market. The product a minimum nitrogen and sulphur concentration of 21% w/w and 24% w/w respectively. Cadmium and mercury impurities are targeted below 10 and 5 ppm respectively. Lead is maintained below 500 ppm. Other nickel laterite operations have marketed some levels of impurities as a trace metal component.

The revenue stream from this by-product is not incorporated into the base case financial modelling.

### 17.3.3 Scandium oxide

While scandium recovery has not been considered in the base case financial model, the process plant considered has been designed to allow the incorporation of a scandium circuit that would produce a 99.9% $Sc_2O_3$ product, which will be dispatched to customers in 25 kg drums or bags. The specification limits of individual impurities will depend on the final product application (i.e. aluminium alloy production or solid oxide fuel cell electrolyte production). A significant amount of scandium oxide can be produced as scandium demand grows. While an average of 170 tpa of scandium oxide can be produced, it was assumed a flat 50tpa was produced as an upside case.

## 17.4 Throughput

A preliminary production schedule was developed using the historical pit optimisation undertaken for the 2005 Feasibility Study Update. These values have been maintained for the October 2016 PFS. The nickel and cobalt tonnage and feed grades are shown below in Figure 17-2. Plant feed tonnage ramps up over three years to the design throughput of 2.5 Mtpa, Production in Year one is 40% of design (1 Mtpa) and 70% of design in Year 2 (2.1 Mtpa). The feed grade is optimised in the early years of operation to maximise the Project's returns.

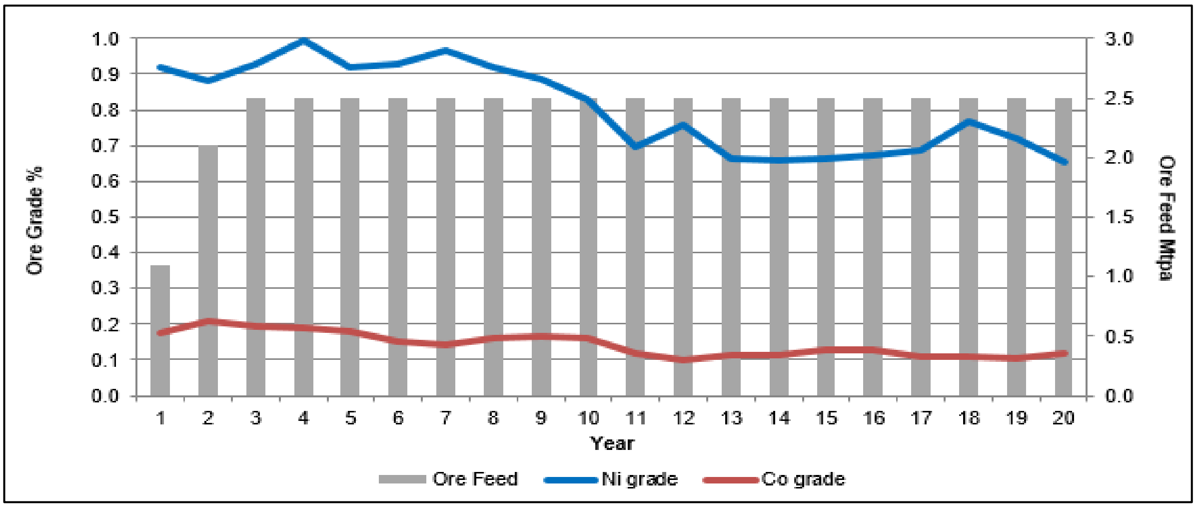



**Figure 17-2: Autoclave feed profile**

The associated nickel and cobalt metal equivalent production is shown below in Figure 17-3.

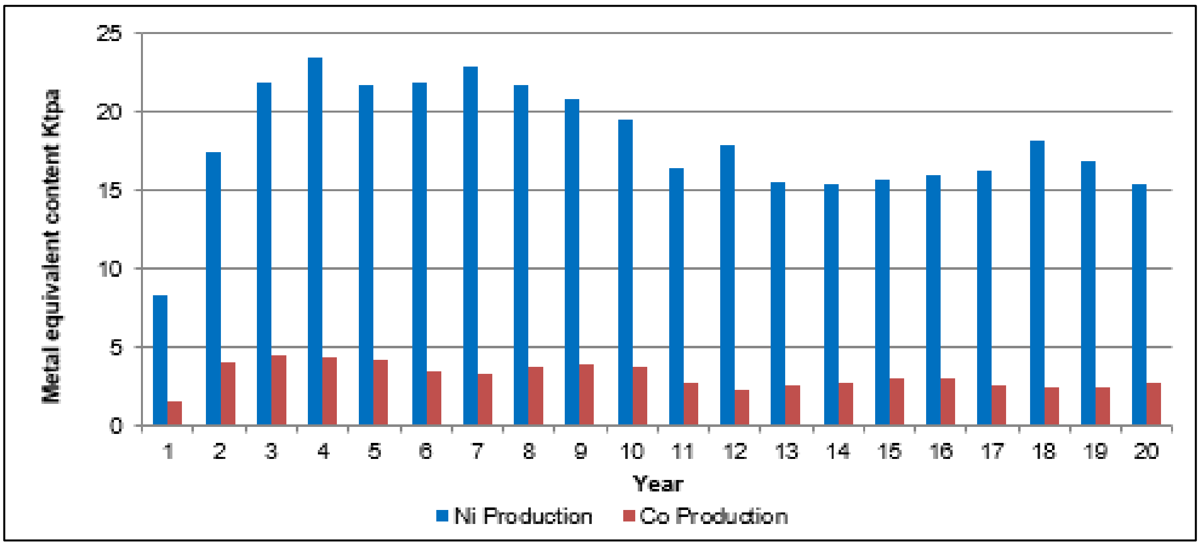



**Figure 17-3: Nickel & cobalt metal equivalent production schedule**

## 17.5 Metallurgical recovery

The key processing recoveries are summarised below in Table 17-3. These values are based on the 2005 Feasibility Study, but updated to reflect the recoveries achieved in the nickel and cobalt RIP circuit which differs from the previous flowsheet. Note that the Feed Preparation circuit recovery of 100% is after early rejection of the low grade coarse oversize fraction.

**Table 17-3: Plant recovery summary**

| Area | Recovery (%) | | |
|---|---|---|---|
| | Nickel | Cobalt | Scandium |
| Feed Preparation | 100.0 | 100.0 | 100.0 |
| Pressure Acid Leach | 96.0 | 95.0 | 86.0 |
| Scandium RIP | - | - | 98.0 |
| Neutralisation | 99.34 | 99.2 | - |
| Nickel/Cobalt RIP | 99.65 | 99.73 | - |
| **Total Plant** | 93.5 | 92.7 | 84.3 |

# 18 Project Infrastructure

The Syerston deposit is situated in central New South Wales, about 450 km (6 hours) drive West North West of Sydney. The Project is well supported by major centres, with the mining communities of Parkes, Dubbo and Condobolin, all located within the vicinity of the Project. The local town is Fifield which is located 4 km from the project area. Access is via a short stretch of minor sealed public roads from Fifield and then by formed private gravel road for the last few kilometres and access to site. The Project Location is shown in Figure 18-1.

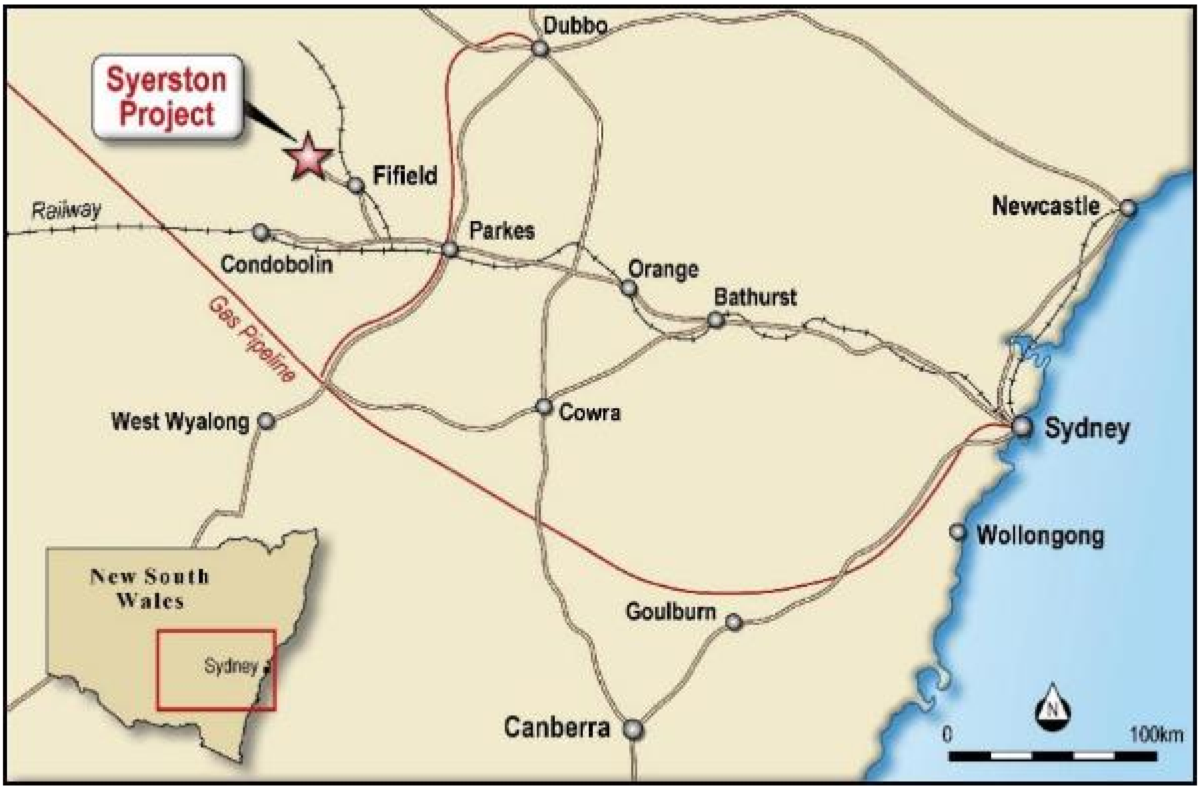



**Figure 18-1: Project location plan**

The proposed Project infrastructure facilities are representative of those required to support a modern 2.5 Mtpa complex hydrometallurgical plant and open pit mining operation. The Project infrastructure comprises the following:

- Access road, internal roads and haul road
- Rail siding
- Integrated power station, high pressure steam boiler and acid plant
- Site buildings - office and administration complex, workshops, stores, ablutions and change house, fences and security
- IT and communications systems
- Sewage plant
- Store and laydown facilities
- Ore stockpiles and waste stockpile area
- ROM stockpile
- Processing plant and associated facilities
- Raw water storage to manage rainfall runoff

- Process plant and mining workshops
- Tailings storage facilities
- Evaporation ponds
- Borefield
- Reverse Osmosis Plant
- Analytical and metallurgical laboratory
- Mobile equipment
- Diesel fuel storage.

These are discussed in further detail in the following sections. A general mine site layout showing some of the key site infrastructure is shown below in Figure 18-2.

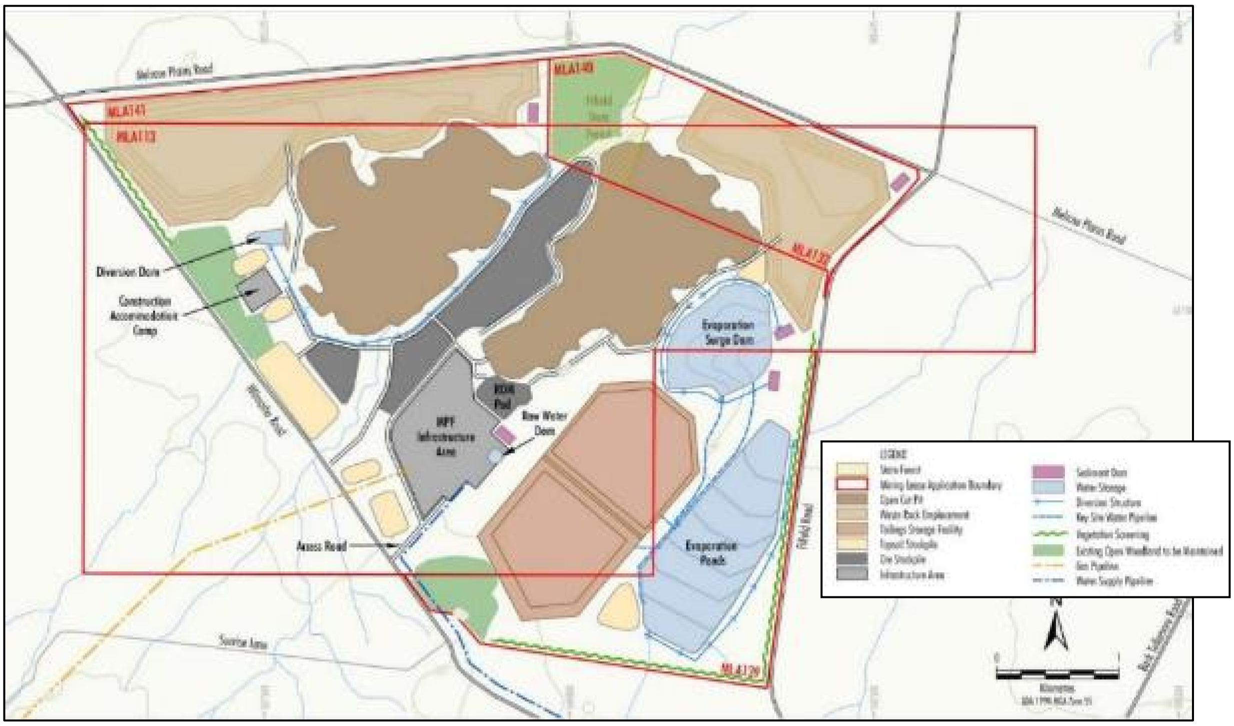



**Figure 18-2: Mine site layout**

One of Syerston's competitive advantages is its proximity to existing infrastructure. The Project is located relatively close to the Moomba-Sydney natural gas pipeline, a rail line is located within 20 km of Syerston, giving it access to the ports of Sydney and Newcastle; and major bituminised arterial roads provide good access to the site. The major city and town centres have excellent infrastructure including transport, airport and rail facilities, all of which are available for project requirements. The Project and associated infrastructure are located within the Lachlan and Parkes Shires and the borefield providing water for the Project is located in the Forbes Shire.

The mining workforce will be predominantly sourced from Condobolin, Fifield, Parkes and other nearby surrounding towns, all of which have a strong historic mining association, with other key technical staff brought in from regional centres as required. There is no site accommodation village. Non live-in employees and contractors would be accommodated in the local towns. Regional public airports at Dubbo and Parkes connect to Sydney, Brisbane, Melbourne and a number of other regional centres with multiple daily flights.

The mining industry within the region is well understood and supported by the major centres. Several large operating mines exist within the region which includes the North Parkes, Peak Hill, Browns

Creek, Lake Cowal and Cadia mines. The local towns' economies have historically been supported by the mining activity within the region. In particular, the former magnesite mine, which forms part of the project area, employed up to 60 local residents. Tin, copper and gold has also been mined to varying degrees within the district. The general response from local government and the community to the re-initiation of the Syerston Project by Clean TeQ has been positive, as evidenced in the Project's Development Consent, originally given in 2001 with subsequent modifications in 2006 and again in May 2017.

The infrastructure design and development has largely been undertaken during the earlier engineering feasibility studies for a 2.0 Mtpa (2000) and 2.5 Mtpa (2005) nickel and cobalt operation completed by the previous owners (Black Range Minerals and Ivanplats Syerston respectively). Work has been continued by Clean TeQ in 2016 and 2017, specifically associated with the application to modify the Project's 'Development Consent' which has now been given. This included Voluntary Planning Agreements (VPA) agreed with the local Shires outlining contributions The Project would make to road upgrades, road maintenance and contributions to community based activities. It also updated other aspects of the Development Consent impacting on local community stakeholders, including a review of several key aspects of the infrastructure including public roads, railway sidings, limestone quarry and the natural gas pipeline.

Because of the limited change in the infrastructure for the Project, the capital cost estimate associated with the infrastructure has been factored (by Clean TeQ) based on a SNC-Lavalin cost escalation assessment undertaken in August 2016. Additional allowances have been made for items excluded in previous studies, specifically to upgrade the local road network. A number of other costs have been excluded as they will be provided on a Build-Own-Operate (BOO) or supply type arrangement. Given the previous level of cost estimate accuracy was at a feasibility level of study with detailed supporting scope, and the revised cost is at a prefeasibility level of accuracy, this is an acceptable revaluation methodology.

## 18.1 Roads

The Project considers site access roads, mining haul roads and other internal roads. A section of dual lane roadway approximately 23 km in length is constructed and used as the public access road to the Syerston mine site. This section of roadway begins approximately 7 km North of the town of Trundle along the Trundle-Tullamore Roadway at the turn-off to Fifield, and continues to the mine site access road.

The dual lane roadway is an upgrade of the existing, sealed single lane roadway to Fifield with the exception of a section of road that may potentially be built to bypass the town of Fifield. The roadway will be constructed to meet all NSW RTA and local government standards for the operation of double road trains and other heavy vehicles. Access to the Project Site shall be by designated haulage routes within the Lachlan, Parkes and Forbes Shires only (no haulage will be allowed on MR354) and this will be governed by a Traffic Code of Conduct.

In addition, a 17 km road upgrade will be required for the dirt bypass road from Trundle to the State Route 90 east of Bogan Gate, known as the Middle Trundle Road (SR83). The upgrade will be the sealing of the gravel sections by the Company and the remainder by the Forbes Shire, with the requirement to meet NSW RTA and Government Standards for heavy vehicles in accordance with Austroads specifications. The local road and rail transport network is shown below in Figure 18-3.

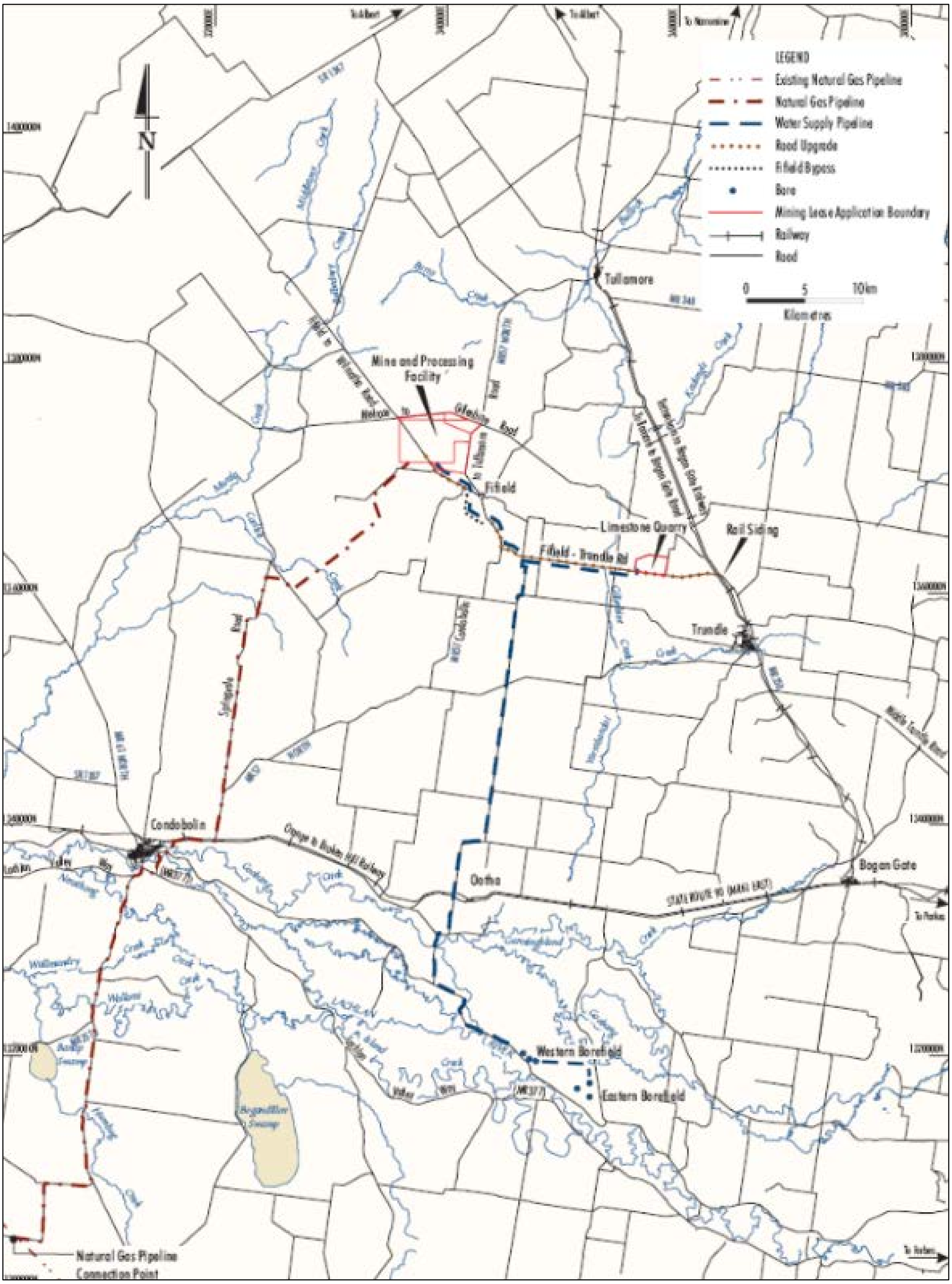



**Figure 18-3: Local transport network**

## 18.2 Accommodation

The closest centre of population to the Project is the small town of Fifield, however a number of larger towns and regional centres that enjoy excellent infrastructure, services and communications are within commutable distance. These towns such as Condobolin, Trundle, Tullamore and Parkes as well as other's further afield serve as mining hubs for other mineral operations. The operation's labour force requirements will be based in these areas on a drive-in/drive-out (DIDO) basis. Any shortfall of sufficiently qualified and experienced personnel in these locations for a relatively complex hydrometallurgical processing facility would be addressed with a DIDO workforce from centres further afield or fly in/fly out (FIFO) workforce if required to supplement the overall workforce.

## 18.3 Airstrip

The Project workforce would be largely based in the local towns and surrounds as normal practice for mining operations in the area. This may be supplemented from further afield for specialist technical and management roles required in a large hydrometallurgical operation. The major regional public airports at Dubbo and Parkes connect with multiple flights daily from Sydney, Brisbane, Melbourne and a number of other regional centres. These provide for any flight requirements.

## 18.4 Water

Previous water investigations by Coffey in 2000 determined that insufficient water was available in the immediate project area to meet the historical plant requirement. The closest viable source of sufficient water was the Lachlan River, approximately 65 km to the south of the project area. Black Range and Ivanplats Syerston completed the EIS and Development Consent assuming establishment of this borefield.

Clean TeQ engaged Golder Associates Pty Ltd in March 2015 to determine the current status of water supply in the area. The report concluded that there were no single groundwater or township water source that could provide the required demand. Therefore, the most practical water source for the project remains to be the established borefield.

The Syerston Plant will obtain water from this borefield and river water supply system some 65 km south of the plant. Bore water will be produced from two fields, the Eastern Borefield and the Western Borefield. Each borefield will have three bore holes each equipped with a downhole pump. Water from the borefield will be pumped into the Water Supply Storage Tank and pumped through a dedicated pipeline to the Syerston site.


**Figure 18-4: Borefield**

The estimated Project raw water requirement as reported in the PFS is ~450 m³/h. This includes, potable water, fire water, high pressure hose down water, mine utility water and plant water for use in the process as well as feed to the water treatment plant producing high purity water for steam production. The term “plant water” is applied to water for use in the process which has been drawn from the raw water tank or from the raw water dam.

A detailed Project water balance has not been updated as part of the PFS however this work is being undertaken as part of the ongoing Feasibility Study. It is expected that there are some moderate water savings associated with the proposed flowsheet. Water usage minimisation has been incorporated into the Project design and is aimed at ensuring a sustainable borefield abstraction rate. The PFS estimated consumption compares with the previous 2005 FS which reported a Project water balance of 497 $m^3$/h.

The Eastern and Western borefields were established assuming a duty/standby arrangement. To date, one bore in each of the Eastern and Western borefield has been developed, with the Western borefield operating (for local use under a water purchase agreement).

Water Bore Licences were granted to Ivanplats Syerston (the previous project owners) to extract 100 /s (360 $m^3$/h; 3.2 GL/year) from the Project borefield by the NSW Office of Water on 13 June 2006. This volume of water was not sufficient to meet the Project’s entire raw demand and was to be supplemented with water direct from the Lachlan River. At the time, approvals were not obtained for this supplementary water source. While the current project is reducing water demand, it is likely that supplementary water will be required.

Clean TeQ plan to finalise the water the water balance as part of the FS before engaging a water broker to secure any additional water requirement.

River water will be supplied via two river water pumps and intake system. Each pump is rated at 720 $m^3$/hr but the intent is not to use them for the full raw water demand. The pump system is designed to be mounted on a concrete pad sufficiently high enough to be above the flood level on the river bank. The pump intake system is designed to minimise intake of river sediments and depth. Due to the level of suspended solids and organics, the river water will be filtered in a backwash sand filter system prior to delivery to the water supply storage tank. Backwash from the filter will be made available to the local landholder via pipeline.

Two raw water transfer pumps will deliver raw water to the Syerston plant site. The plant will receive an estimated maximum of 10.8 ML/d (450 $m^3$/h) of raw water from the borefield and/or river via a cement lined 65 km 450 mm nominal bore (NB) delivery pipeline. Raw water will also be supplied to the limestone quarry via a mild steel 16 km, 150 mm NB spur line off the main raw water delivery pipeline.

The raw water will be received at the plant site into a single fixed roof raw water storage tank. In addition to the raw water storage tank a raw water storage dam of capacity 30,000 $m^3$ is provided.

Raw water for make-up to the potable water tank is withdrawn from the delivery pipeline upstream of the raw water storage tank.

Most of the demineralised water from the Demineralised Water plant (Reverse Osmosis treatment) is used for raising steam and de-superheating superheated steam. In order to reduce overall water demand air cooling is applied in the Power Plant and the Sulphuric Acid Plant. Water recycled from the tailings storage facility (TSF) and evaporation pond will be managed in a separate storage and pumping system.

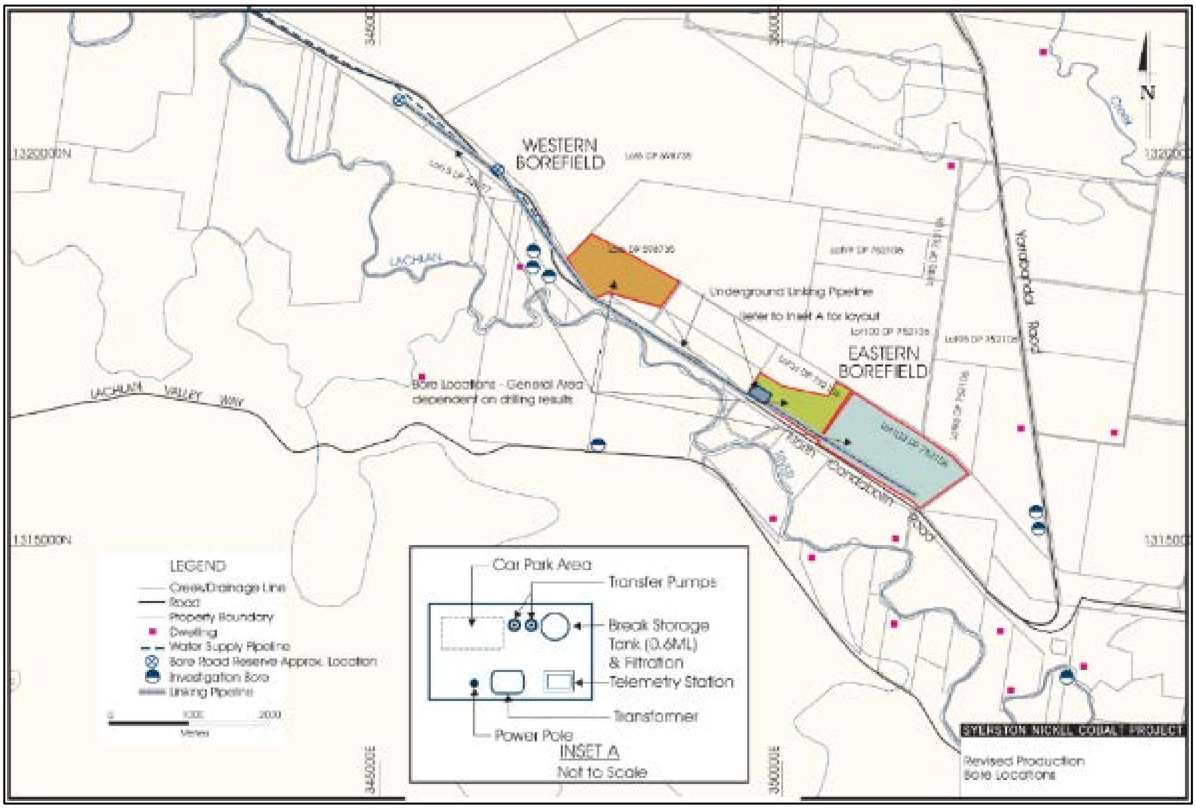



**Figure 18-5: Borefield layout**

Water quality is an important consideration to laterite processing plant design. Materials of construction, for example the titanium clad autoclave, have been selected to meet the requirements of a modest TDS water source. The analysis of the bore water has been taken as the design case as it represents the worst case for TDS and chlorides. The TDS (Total Dissolved Solids) of the water varies from bore to bore. A worst-case design TDS of 870 mg/L has been used which represents the TDS of the Western Borefield as measured by Coffey Geoscience; the associated chloride ion level is measured at 320 mg/L (Chloride ion level in river water is measured at 182 mg/L).

## 18.5 Power and steam

### 18.5.1 Turbine generators

Two gas turbine generators (GTG) units are proposed, each of which comprises a gas generator with combustion system and a power turbine, mounted on a base frame.

Each gas turbine has a dedicated lube oil system. The lube oil tanks are fitted with heaters to heat up lube oil if it is cold. Once the gas turbine is running hot, return lube oil is cooled in a water-cooled lube oil cooler. To ensure that lube oil continues to be fed to the gas turbine in the event of a power failure, a lube pump back-up power system is provided. This comprises ventilated lead acid batteries capable of delivering sufficient power via frequency converters to the lube oil systems of both gas turbines and the steam turbine.

The gas turbine is fitted with an electric starting and barring system to commence the rotor rolling to minimise the inertia to be overcome during turbine run up.

The air intake system is equipped with a baghouse filter to ensure dust free air is fed to the combustion section of the gas turbine. Air jet pulsing automatically cleans the filter bags. The air intake system will also be equipped with an evaporative cooling system to allow inlet air to be cooled so as to overcome loss of performance under hot ambient conditions.

The gas turbine is controlled by a dedicated control system. This system controls the firing of the gas turbine (and the steam turbine and boilers; described below under power plant control system). The compressor stage is a 14-stage axial flow compressor fitted with two variable guide vane rings which are AC servo driven. Resetting the vane position allows optimising of the energy efficiency of the turbine at varying load conditions.

### 18.5.2 Heat recovery steam generators (HRSG)

Each Gas Turbine is fitted with a Heat Recovery Steam Generator (HRSG). This is an insulated duct which is equipped with three sets of pipe water tube pipes to extract heat out of the flue gas into water. The Gas Turbine exhaust gas temperature is 500ºC. The temperature at the exit of the Economiser is 150ºC. The HRSG duct is fitted with several natural gas burners that boost the steam raising capacity by an additional 22 t/h. The burners are controlled by a PLC based burner management system that is part of the control system. A purge air fan is provided for the pre-ignition purge required by the burner management system.

A diverter valve is fitted in the gas turbine exhaust gas ductwork, which can direct gas to a bypass stack. This allows some hot gas to be diverted in cases where steam demand is relatively low.

### 18.5.3 Steam turbine generator set

The waste heat recovered in the HRSG as steam is used in the process plant and excess is used to produce electrical power. This excess steam is directed to the steam turbine generator. In the situation where one of the gas turbines is not available, the steam turbine makes up a large part of the electric power deficit.

### 18.5.4 Auxiliary boiler

The HPAL autoclave operates at an elevated pressure of 4500 kPa and elevated temperature of 250ºC. To achieve this temperature, high pressure steam is added directly to the autoclave. This steam is produced by a gas fired boiler.

The auxiliary boiler is a natural gas fired standalone package boiler that is capable of generating high pressure (HP) superheated steam. This is a conventional boiler comprising an; economiser, evaporator and a superheater. The auxiliary boiler supplements steam supply when one or both of the gas turbines are unavailable or constrained to run at low rates. An auxiliary boiler is specifically required during HPAL system heat-up when overall power demand is low but steam demand is at a maximum.

### 18.5.5 Black start and emergency power generation

Three diesel driven generators are provided and are referred to as the emergency generators. These emergency generators are permanently tied in to the Syerston main power bus. Diesel is fed to the emergency generators from the diesel storage tank. Each emergency generator will consume diesel at a rate of approximately 250 - 300 litres per MWh.

## 18.6 Tailings storage facility and evaporation pond

The final slurry/solution after metal recovery is neutralised with limestone and sent to a tailings storage facility (TSF) and an evaporation pond. The tailings storage facility, the evaporation ponds and the evaporation surge dam are to be located in the south-east corner of the site. The site earthworks contractor will be responsible for the construction of the evaporation surge dam, the tailings dam and the evaporation ponds. Tailings slurry from the process plant will be pumped and distributed to the TSF, which is immediately to the east of the process plant. An evaporation pond is incorporated into the design to reduce excess water requirements due to the positive water balance and the need to bleed liquor to maintain several elements below their solubility limit.

The TSF will cover an estimated total area of 217 ha and be divided into two cells, designated south and north cells. The cells will be progressively raised in a downstream direction to store the tailings for a period of 20 years at a design capacity of 3.35 Mtpa.

Supernatant and rainfall run-off water will be recovered from the surface of the TSF and decanted into a series of ponds and a dam for evaporative disposal. Owing to the water chemistry of the tailings slurry liquor, and the effect shown in testwork on nickel and cobalt extraction, and saturation of liquors with impurities such as calcium and magnesium which will precipitate out as sulphates and other scale, there is minimal opportunity for recycling this water through the process plant. However, the opportunity will be reviewed again during operations.

The evaporation ponds, which lie between the TSF and the eastern property boundary, will cover an estimated total area of 121 ha and be divided into seven ponds of varying elevation, constructed down the natural topography.

The evaporation surge dam, immediately to the north of the TSF, will cover an estimated area of 56 ha, and will be subdivided into four evaporation ponds.

## 18.7 Natural gas

Natural gas is transported to site through a natural gas pipeline. Gas is used both for energy to heat water into HP steam used in the leaching process as well as for electricity, via a gas fired generator. Natural gas would be supplied through a build, own and operate (BOO) contract, including building the gas pipeline lateral and any required booster stations and the final decompression and measuring station at the plant site.

The pipeline branches off from the Moomba-Sydney natural gas pipeline, passes near Condobolin and runs north to deliver natural gas to a metering station at the SW corner of the Syerston process plant site. From the metering station, natural gas is letdown and distributed to the plant.

The current natural gas market on Australia’s eastern states is tight on the supply side and as a result there is upwards pressure on pricing. There is a current moratorium on new land based conventional and unconventional supplies but there is also a current Federal Government review of gas supply options and a current mandate to introduce legislation for the reservation of natural gas exports to ensure sufficient domestic gas supply. The general advice is that insufficient gas availability will not be a risk to the Syerston Project but pricing is less certain and is at risk of escalating prices.

The Project is not sufficiently advanced to enter into formal negotiations for the supply or transmission of natural gas. Due to the quantity required, it is more likely that supply will be through a retailer/gas aggregator rather than the primary producer. Gas suppliers such as AGL, Alinta and Origin have confirmed that there is sufficient capacity in the pipeline main to supply the Project, which is in the order of 2.4 TJ/day of firm (not interruptible) supply and the APA Group (the owners and operators of the pipeline) have been re-engaged regarding the new pipeline and have confirmed that they are able to build and operate it but are yet to provide pricing, which will be based on the natural gas demand established in the FS engineering work.

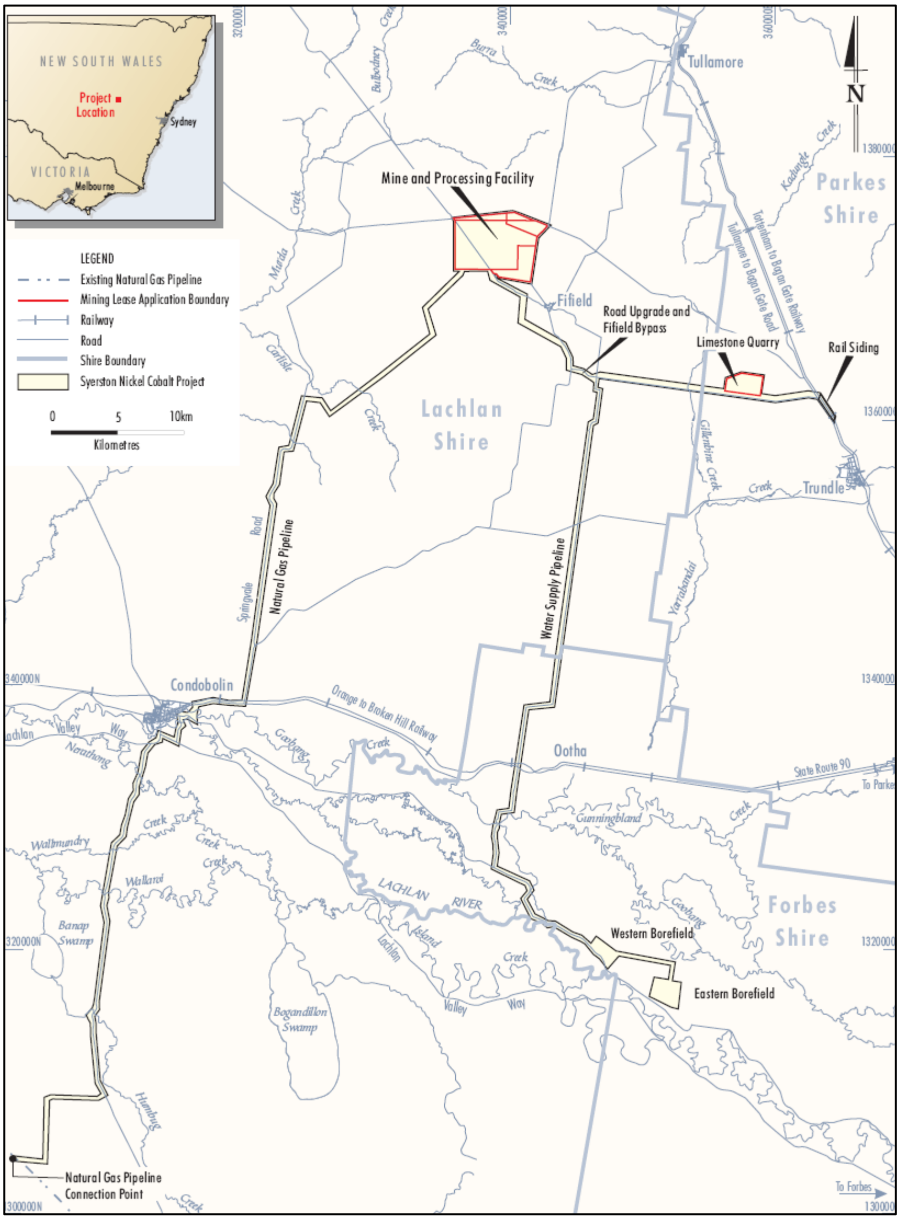



**Figure 18-6: Natural gas pipeline and water pipeline routes**

## 18.8 Limestone

Limestone is required to neutralise process slurries and liquors following acid leaching. In order to meet this requirement, the original FS approvals were obtained to mine a maximum of 600,000 tpa of limestone from the Gillenbine Limestone Deposit, situated approximately 20 km south-east of the Project.

The approved design for the Quarry presented in the EIS includes the removal and stockpiling of waste rock and limestone extraction using conventional open-pit drill and blast methods. Waste rock and low grade limestone would be deposited in an emplacement surrounding the open pit. Figure 18-7 shows the approved Quarry during Year 5. ROM high grade limestone, would be crushed at the Quarry site prior to transport to the Syerston processing facility site.

The crushed high-grade limestone is approved to be transported to the MPF site via the Fifield to Trundle Road, the proposed Fifield Bypass, and the Fifield to Wilmatha Road using side tipping road trains. Loading and haulage of crushed limestone to the MPF would be conducted five to six days/week, 52 weeks/year. Mining and crushing operations would be undertaken on day shift only.

Ancillary infrastructure approved to be constructed at the Quarry includes site offices to be located at the existing “Westella” homestead, workshops and maintenance facilities. The existing electricity supply to the “Westella” homestead would be utilised at the site offices and workshops. Industrial electrical requirements of the crushing facility would be provided by a diesel powered 500 kV generator set.

Comprehensive mining and engineering work is based on the 2005 FS and allows for a detailed tender scope of work. Because of the PFS level of study, no formal agreement has been entered into for the contract operation of the limestone quarry although the identification of suitable contractors has begun and some preliminary discussions have been undertaken with potential providers of this service. It is reasonable to expect that this work could be done by a number of contractors and any tender for this work would be competitive. It is likely this contract would be incorporated into the mining contract.

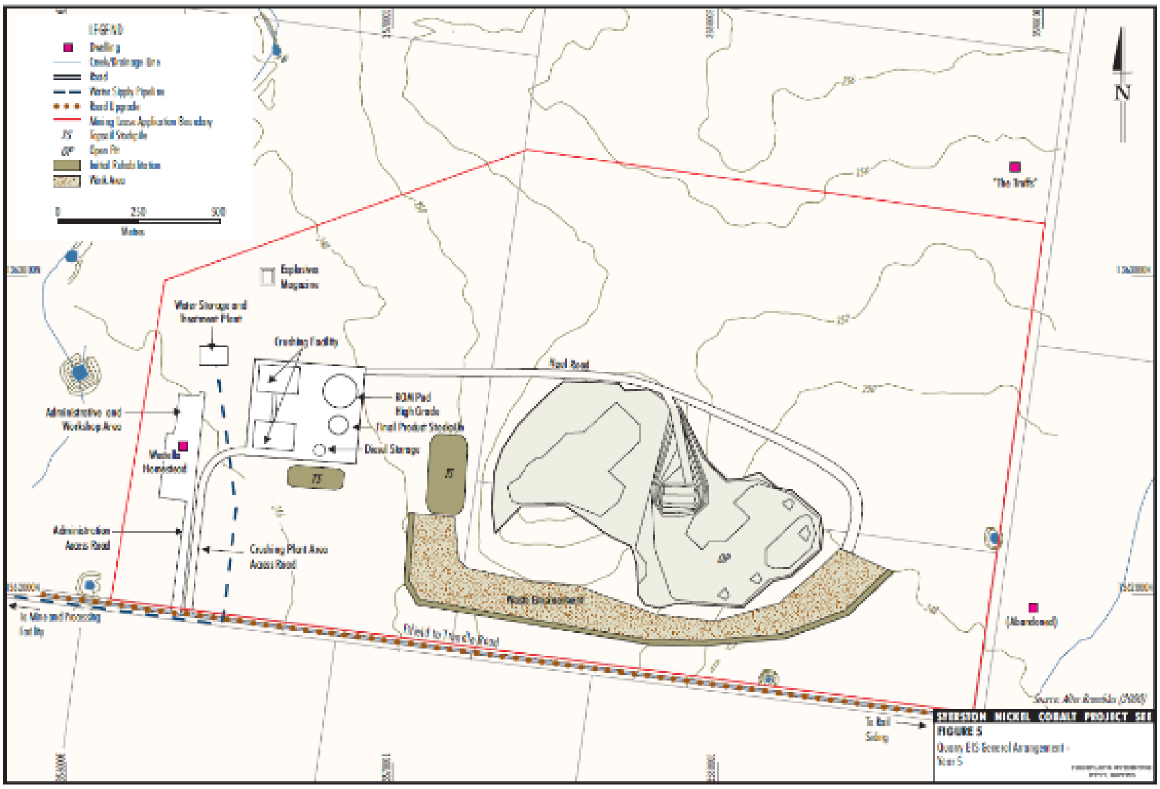



**Figure 18-7: Limestone quarry general arrangement**

## 18.9 Rail and Port

Transport logistics are critical to the operation, particularly the transport of bulk sulphur required for sulphuric acid production. A rail line exists within 20 kilometres of Syerston, giving it access to the ports of Sydney (Port Botany) and Newcastle, and bitumen roads providing good access to the site. A rail siding just north of the town of Trundle has been designed to accommodate these deliveries as shown below in Figure 18-8.

It is envisaged that the siding would be constructed and operated by a third-party service provider specialising in this area. The design is well developed as a result of the 2005 FS and allows for a detailed tender scope of work. Because of the PFS level of study, no formal agreement has been entered into for the construction of the siding or its operation, although some early investigations have been undertaken with potential providers. Preliminary discussions have been held with John Holland, the above rail (operators of trains and rolling stock) regional operator of the railway regarding operation and logistics. Discussions have not yet been held in respect to the below rail (track management) for a new siding and rail to site.

Sulphur, is imported in bulk, and transported to site via rail/road. Limestone will be mined from the Company's Westella limestone quarry and trucked to site. All other chemicals and general freight will be trucked or railed to site in either B-Doubles, ISO containers, IBC's or bulk bags and stored on site in a dedicated reagents area.

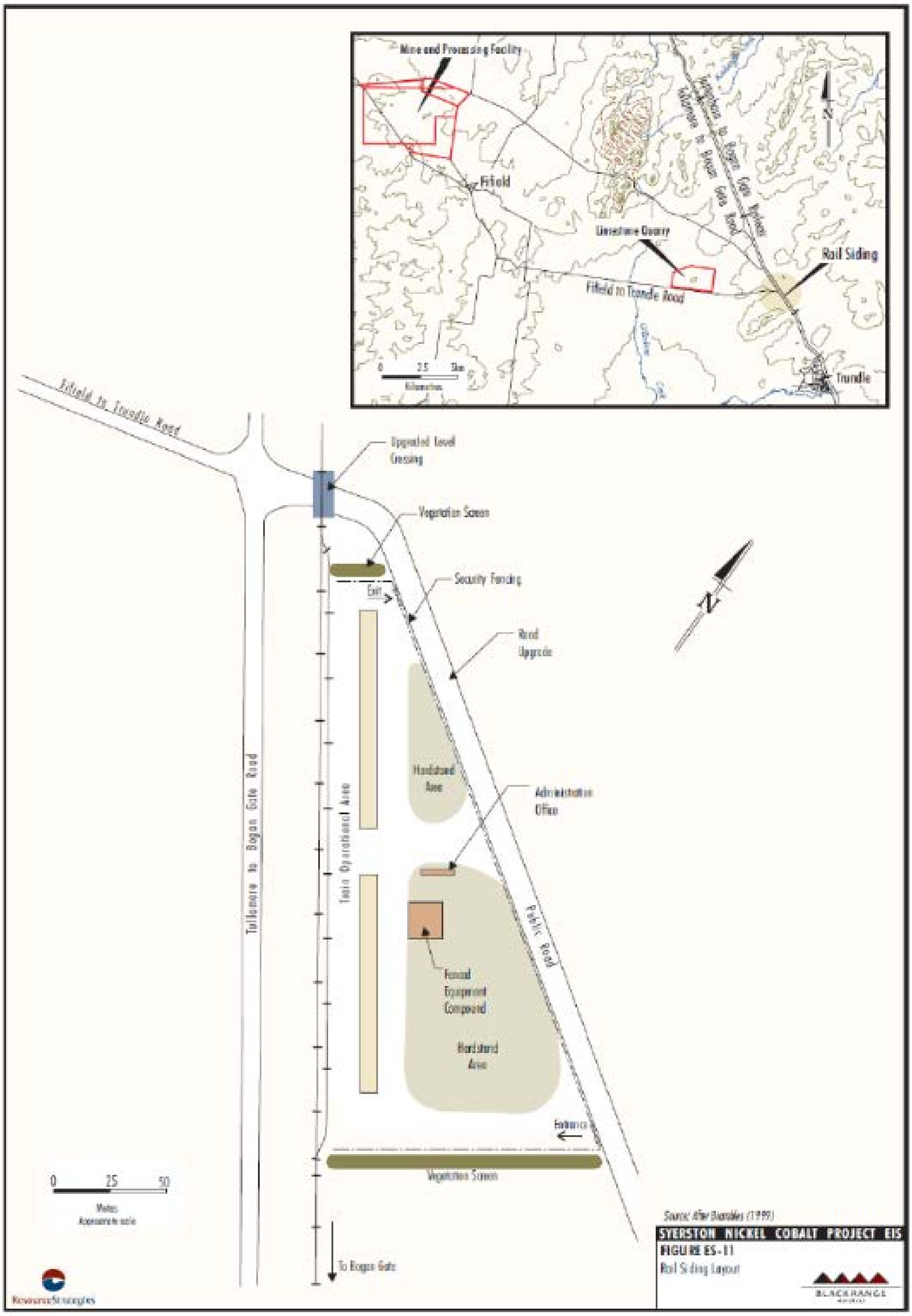



**Figure 18-8: Rail siding layout**

## 18.10 Site buildings

There are six site buildings provided for on the Syerston processing plant site. These are:

- The operations building which houses administration, management and technical support staff
- The central control room building which houses the distributed control system console room, data processing facilities, offices for production supervision personnel and crib facilities for production technicians
- The workshop building which houses maintenance supervision and planning, personnel, maintenance work area, tool store, and dispatch area for rotable spares
- The warehouse building which houses stores and procurement, personnel, storage areas, goods receivable area and stores issuing areas;
- The change rooms that house dirty/wet and clean change areas, ablution areas, laundry collection and distribution areas
- Analytical and metallurgical laboratory facilities.

All the buildings are air conditioned and are equipped with crib and ablution facilities.

During construction, a temporary building of 900m$^2$ will be provided to house Syerston advance personnel and process commissioning teams.

## 18.11 IT and communications

The intent is for the Project's IT and communications infrastructure requirements to leverage off existing regional communications infrastructure as much as possible in order to reduce the capital expense whilst addressing the core business communications requirements. IT content considers the Syerston mine site, process plant, infrastructure, haul road and linkage back to the Clean TeQ's head office.

Open telecommunication will be established incorporating regional networks. Telstra currently provides 3G mobile services in the general area however service can be limited, although antennas can be used to boost coverage and/or data speed. Satellite services are likely to provide the initial communications services to site during the project development phase of work, and will continue to be used as back-up. Satellite service provider options provides an alternative if there is no wide area network (WAN) solution. The long-term option for permanent communication to the site will be provided using a private single microwave system to the nearest Telstra optic fibre network access point at Fifield.

The broad functional scope requirement for IT and communications is based on the site service requirement focusing on the following:

- Public network (Telstra) IP (phone and data) connection and services
- Telstra mobile 3G access
- Corporate voice/phones
- Corporate data services/network communications including Wide Area Network (WAN) and Local Area Network (LAN)
- IT Network equipment including server (hardware, operating system and storage)
- User hardware including desktops, laptops, printers
- Cameras/feeders/video
- UHF/VHF two-way radio system
- Wireless

- FM radio re-broadcast
- Communications towers at the processing site (with mine site coverage)
- Communications shelters, rooms and power supplies.

A clear VHF radio path for the mine to the plant will be established along the haul road using a single central communications tower situated at an elevated location on the site tenements. If necessary, a repeater station will be installed at the mine to ensure full coverage. Communications into the administration area will be carried by cable from the plant site tower.

A VHF radio system will be established for the plant and mine sites with coverage across all areas of the operation. The radio system will cover multiple channels. The communications package capital estimate includes an allowance for the supply of VHF radios for fixed, mobile and hand-held purposes and a base and handheld radio for air traffic communication.

Intra-site connections (process plant and mine are separate entities) will be through an internal Local Area Network Ethernet system. All inter-site links as part of the Local Area Network is provided via satellite network.

## 18.12 Mobile equipment

The bulk of the mining vehicles will be provided by the mining contractor. Plant and general administration light vehicle and mobile equipment allowances have been made including; light vehicles including plant and maintenance vehicles utilities, managers' vehicles, modified fire service and ambulance vehicles, ROM loader, skid steer loaders, store forklift, flat bed maintenance truck with hoists, small mobile crane and other miscellaneous vehicles.

An allowance has been made for these in the operating costs as they are assumed to be leased. There is currently a strong second-hand mining vehicles market and the purchase of second hand near new vehicles may be an opportunity to reduce costs.

## 18.13 Infrastructure third party supply & operation

The following facilities are expected to be provided on a BOO basis or direct supply cost, and are not incorporated into the capital cost estimate. It is likely that minimum payments or take-or-pay agreements will be used. This will impact on early operating costs over the first two years as production ramps up to target levels.

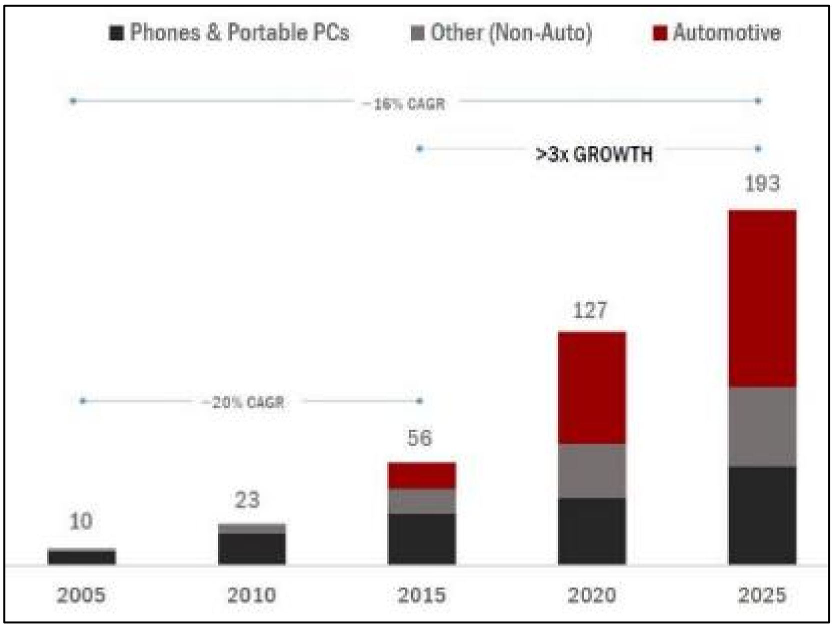
- Mining operation including fleet, buildings and workshops
- Limestone from owner's quarry based on a per tonne supply cost
- Supply of all reagents, including sulphur port and rail facilities
- Liquid nitrogen plant
- Mobile plant and equipment required for operations
- Site mobile vehicles plant
- Natural gas pipeline.

Re-engagement and initial discussions have been held with a number of the key third providers for their supply however the technical details and commercial terms have not been finalised, largely due to the level of study currently being undertaken. In most cases this leverages the engineering work currently undertaken during the 2005 Feasibility Study. Firmer pricing will be developed during the next phase of study during more formalised negotiations.

# 19 Market Studies and Contracts

## 19.1 Li-ion batteries driving demand for nickel and cobalt

A tipping point has been reached in Li-ion battery market through the accelerated growth of EVs. There has been a 20% growth rate in Li-ion batteries in the last decade, and while forecasts vary, this level of growth is expected to continue into the future, driven by the increased adoption of EVs globally. This growth is due to increasing fuel efficiency targets and carbon dioxide limits set globally. With the significant reduction in battery costs and increasing energy density, EV's and hybrid vehicles represent the logical solution for the industry to meet these targets.



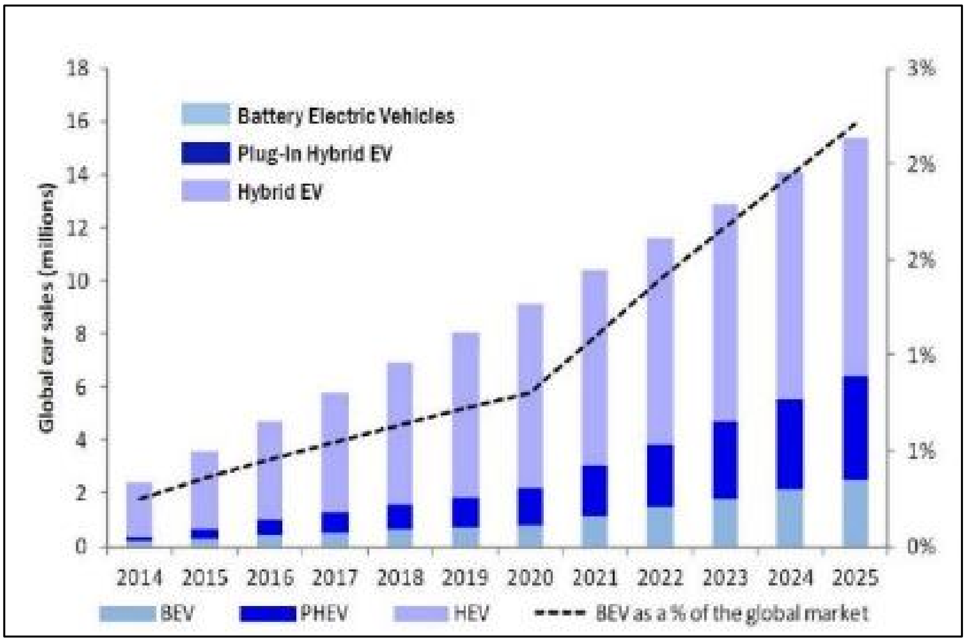




**Figure 19-1: Historic and forecasted lithium batteries sales (GWh)[1] and EV[2] sales**

[1] Avicenne Energy Analysis 2014
[2] Deutsche Bank research, 2016

While Tesla has historically been the most active in the EV space, large car companies such as Daimler, VW, BMW and GM are beginning to make significant investments. Similarly, while the US has previously led in EV sales (100,000 in 2015), this has now been surpassed by China, who sold over 300,000 EVs in 2015 [3].

### 19.1.1 Cathode chemistries of choice

Li-ion cells contain a positive and a negative electrode. The positive electrode (cathode) is made of various formulations or 'chemistries' of oxidized metals. The negative electrode is generally made of carbonaceous material (natural and synthetic graphite). When the battery is charged, ions of lithium move through an electrolyte from the cathode to the anode and attach to the carbon. During discharge, the lithium ions move back from the carbon anode to the cathode.

The different battery types or 'chemistries' are defined by the compositions of their metalliferous cathodes. There are five main battery chemistries which comprise the majority of the lithium batteries market. Of those, lithium-cobalt-oxide (LCO) is the dominant battery in portable electronic devices. The nickel-cobalt-manganese (NCM) and nickel-cobalt-aluminium (NCA) chemistries are increasingly becoming the industry standard for electric vehicle applications, due to their high energy density.

In recent years, China's automotive industry has favoured adoption of lithium-iron-phosphate (LFP) and lithium-magnesium oxide (LMO) battery chemistries. However, there is a clear global trend to the adoption of NCM and NCA chemistry due to their higher energy densities, increased life cycle and the auto industry's preference for passenger vehicles with longer range. Significant growth in the lithium batteries sector is expected to come from NCM and NCA chemistries, both of which can contain relatively high nickel and cobalt content.

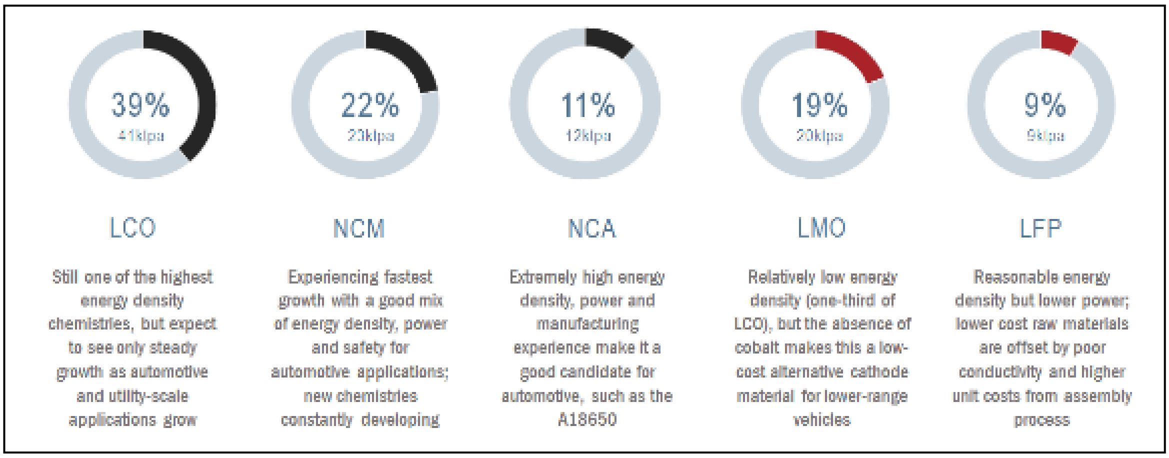



**Figure 19-2: Lithium batteries chemistry market share[4]**

Nickel and cobalt are critical to the dominant chemistries of the lithium batteries industry. Depending on the cathode and chemistry used, the combination of nickel and cobalt can represent up to 80% of the cathode raw material cost and 20% of the total battery cell cost. This makes the lithium batteries industry's cost of production susceptible to these metal markets.

[3] Goldman Sachs – Charging the Future: Asia leads drive to next-generation EV battery market, 2016
[4] *Source: Avicenne Energy Analysis 2014*

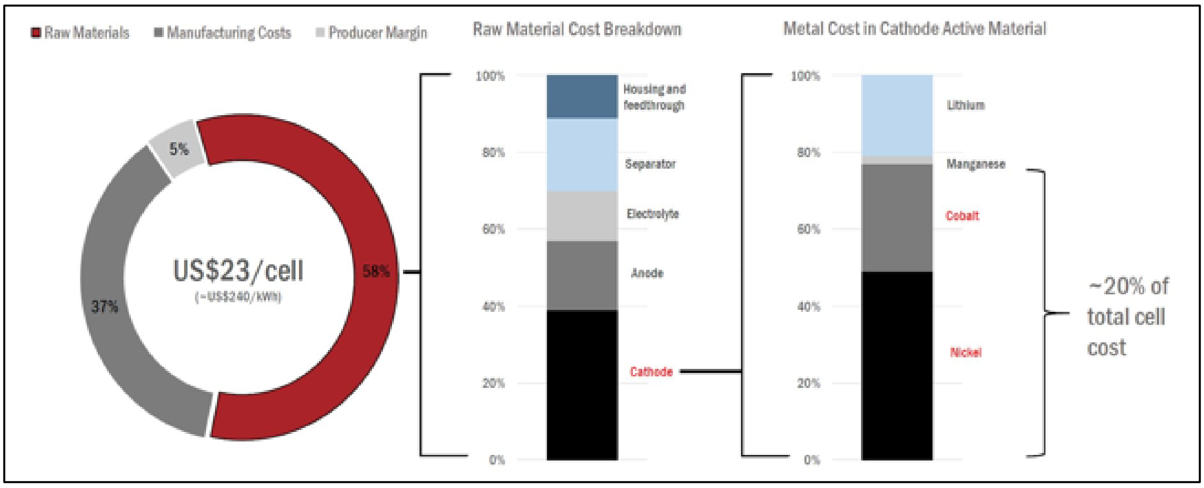



**Figure 19-3: Raw material cost breakdown in batteries [5]**

### 19.1.2 Nickel and cobalt critical for supply

lithium batteries cathode production requires high purity precursor materials to ensure high performance and extended battery life. NCA and NCM battery chemistries require high purity nickel sulphate ($NiSO_4.6H_2O$) and cobalt sulphate ($CoSO_4.7H_2O$) to produce precursor materials. LCO battery chemistry requires cobalt oxide.

The predicted growth in the lithium batteries market means that a considerable amount of high purity nickel sulphate and cobalt sulphate will be required over the next ten years. As such, reliable and cost-competitive nickel and cobalt supply has an important role to play in the future of lithium batteries's.

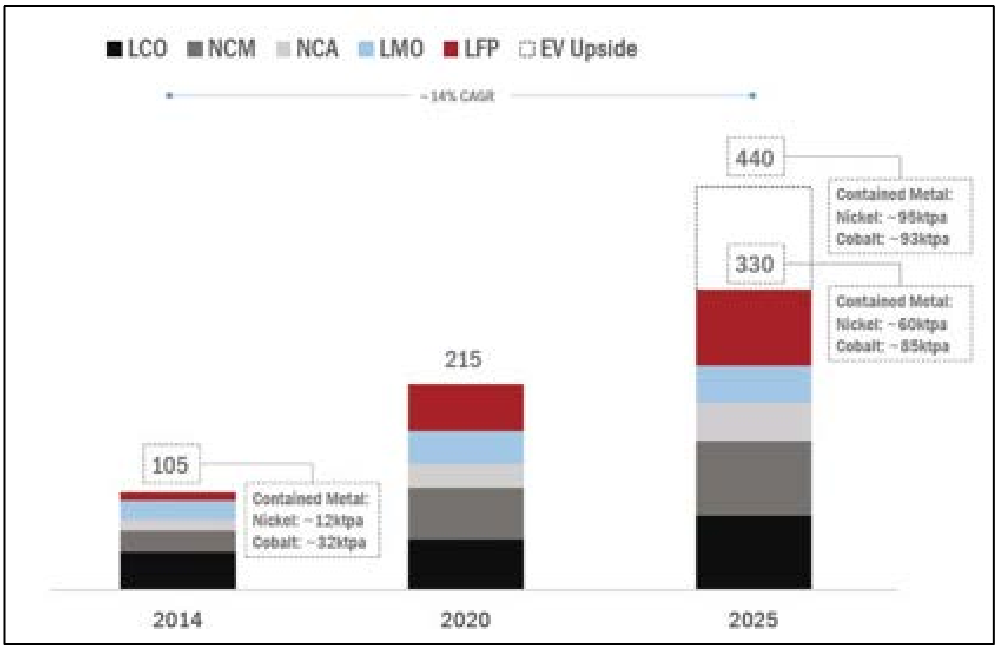



**Figure 19-4: Cathode raw material demand ('000 tonnes) [6]**

[5] Roland Berger (2012) and internal analysis. Assumes a 96Wh PHEV cell (26Ah, 3.7W) using NCM622 cathode chemistry. Cathode cost includes non-metallic materials (carbon black, binder, foil).
Internal assumptions concerning split of costs assumes average long-term prices of Ni USD7.00/lb; Co USD12.00/lb; Mn USD1.00/lb; Li USD6.50/kg (as LCE).
[6] Avicenne Energy Analysis 2014. EV Upside based on Avicenne upside case for 2025 of 2.6m units of EV sales. Metal demand based on internal company estimates

While there is a large and established market for nickel which is driven by the global steel sector, almost all of the world's cobalt is produced as a by-product from nickel and copper mines. For this reason, cobalt stands apart as one of the few metals consumed at industrial-scale that has almost no source of primary supply. Global refined production in 2015 was in the order of 90,000 t of contained cobalt, a large portion of which was exported to, and processed in, China. In order to meet the demands of the growing lithium batteries market, there will need to be a significant increase in global supply of cobalt. At a time when nickel and copper prices are at or near long-term historic lows, this presents real challenges for cobalt supply, as seen in recent or pending mine and refinery closures in Africa (Katanga Mining) and Australia (QNI).

### 19.1.3 Sustainable and auditable supply of materials

In addition to the risk through by-product dependence, global cobalt supply is heavily concentrated in the Democratic Republic of Congo (DRC). In 2015 production sources in the DRC represented 65% of global mined cobalt supply. A large portion of this production was from artisanal mining operations involving child labour.[7] While cobalt is not listed as a 'conflict mineral', the lithium batteries industry is under increasing pressure to demonstrate an auditable cobalt supply chain to ensure that responsible procurement practices are adopted.

A recent report by Amnesty International and Afrewatch, "*This is what we die for: Human rights abuses in the Democratic Republic of the Congo power the global trade in cobalt*", highlighted the child labour practices adopted in many of the artisanal mines and urged the global electronics and automotive industries to provide better auditing of their supply chains.

Syerston provides a unique opportunity for the lithium batteries industry to source a significant amount of cobalt with a fully auditable supply chain.

## 19.2 Syerston produces lithium batteries products direct from mine to industry

### 19.2.1 Dedicated lithium batteries products

The conventional process route for nickel and cobalt sulphate production is outlined below. Nickel ore, also containing low concentrations of cobalt, is mined and processed on site to produce either metal or an intermediate concentrate, typically a mixed hydroxide, sulphide or separate briquette product.

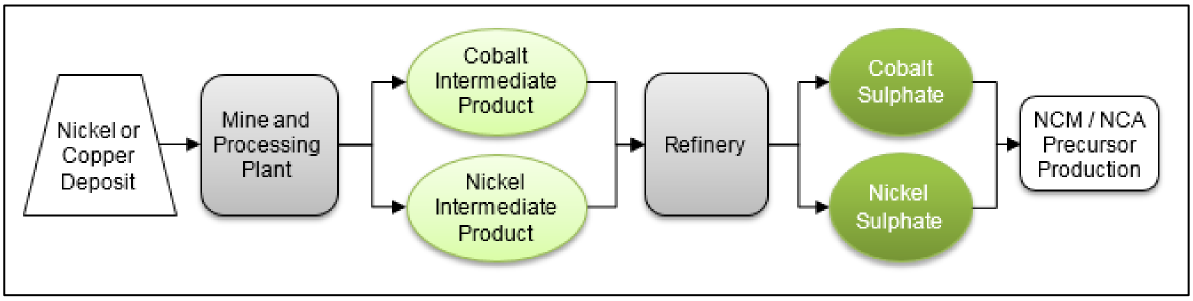



**Figure 19-5: Conventional nickel and cobalt sulphate production process**

The production route for nickel is driven by the steel industry, which consumes over 90% of nickel produced. Only small portion (currently 5%) is converted to sulphate products used in electroplating and lithium batteries. This portion is sent to a separate refinery, where it is re-leached with acid under

[7] *Darton Commodities, "Global Cobalt Review, 2015-2016"*

pressure, purified and then crystallised to produce nickel sulphate. Similarly, cobalt intermediate products are sent to refineries where they are re-leached, purified and crystallised to produce cobalt sulphate. This is an inherently inefficient supply route, as:

- Nickel and cobalt is precipitated and then re-leached, increasing the cost of sulphate production;
- Often the mines and refineries are separate companies, meaning there are additional costs due to margins and transport; and
- Particularly with cobalt, its production is dependent on nickel and cobalt demand and therefore is unlikely to drive an increase in minable production without a significant increase in the cobalt price.

Syerston's nickel and cobalt production process produces nickel and cobalt sulphates in the processing plant, providing a unique opportunity to exclusively produce products required for the lithium batteries industry.

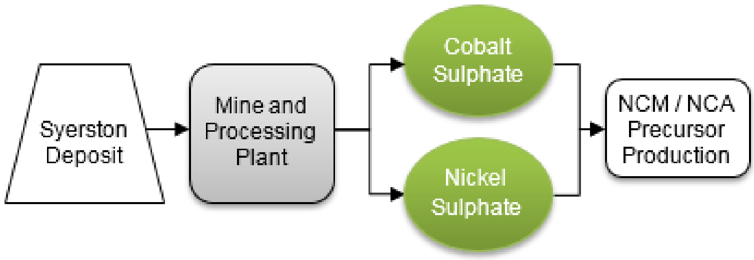



**Figure 19-6: Syerston nickel and cobalt production process**

The use of Clean TeQ's RIP process extracts nickel and cobalt directly from leached slurry, producing sulphates as a part of the process. This allows for purification and production of separate sulphate products on site, without the need for a separate refinery. The benefits of this approach are:

- Direct production of nickel and cobalt sulphate products, without the need to precipitate and re-leach, lowering the total operating cost
- Cobalt is sourced from one mine, providing complete transparency of production forecasts
- Other technologies, such as solvent extraction have the ability to follow a similar process route, but have the following disadvantages:
  - Solvent extraction can only be used on clarified liquors, not slurries. Therefore, a solid/liquid separation step is required (CCDs), which, based on previous studies completed at Syerston, increases the capital cost by 8%-10%
  - The use of CCD's dilutes the nickel and cobalt, increasing the size of the solvent extraction circuit
  - There are soluble metal losses in the underflow of the CCDs, which lower nickel and cobalt recovery
  - The concentrations of nickel and cobalt are relatively low, reducing the efficiency of solvent extraction, increasing capital and operating costs.

Use of RIP eliminates all of these issues. It also allows solvent extraction to be used in purification, at a much smaller scale at nickel and cobalt concentrations much higher and therefore more suited to the process.

## 19.3 Product pricing

Both the nickel and cobalt markets have established trading platforms, either through the LME or Metal Bulletin. However, sulphate product pricing tends to be more opaque, due to the relatively small size of this sub-market. Sulphate pricing mechanisms are typically determined on a premium to either LME (nickel) or Metal Bulletin (cobalt), reflecting the additional cost to convert nickel and cobalt bearing materials into high purity sulphates. These premia can often be high but do appear to vary substantially depending on the market and product quality.

Prices for nickel and cobalt metal have decreased in the last few years, driven by a slowdown in growth in China, principally the steel industry. Importantly this decrease in metal pricing has occurred coincidentally with the increase in demand for Li-ion batteries. Li-ion battery production costs have steadily fallen over the past decade. While this reduction is partially accounted for in the gradual increase in battery production capacity, a portion is due to the decrease in raw material prices.

Until recently, the lithium batteries industry has not been a significant consumer of nickel and, more importantly, cobalt. However, this is likely to change in the future. As production capacity is filled, raw material costs, particularly nickel and cobalt, will become much more important in the total battery production cost. Cathode costs represent the largest raw materials cost in the production of Li-ion cells, with nickel and cobalt costs far exceeding lithium costs in the production of LCO, NCM and NCA cathodes.

Supply of critical metals for the battery sector, particularly cobalt, will require new sources of production to keep up with growing demand from the Li-ion battery industry.

Recent long-term metal price forecasts from market analysts forecast nickel and cobalt prices to rise over coming years, finding equilibrium again at historic long-term average prices. These forecasts predict nickel prices to rise to between USD8-10/lb and cobalt prices to increase to between USD15-17/lb by the time Syerston is estimated to come online.

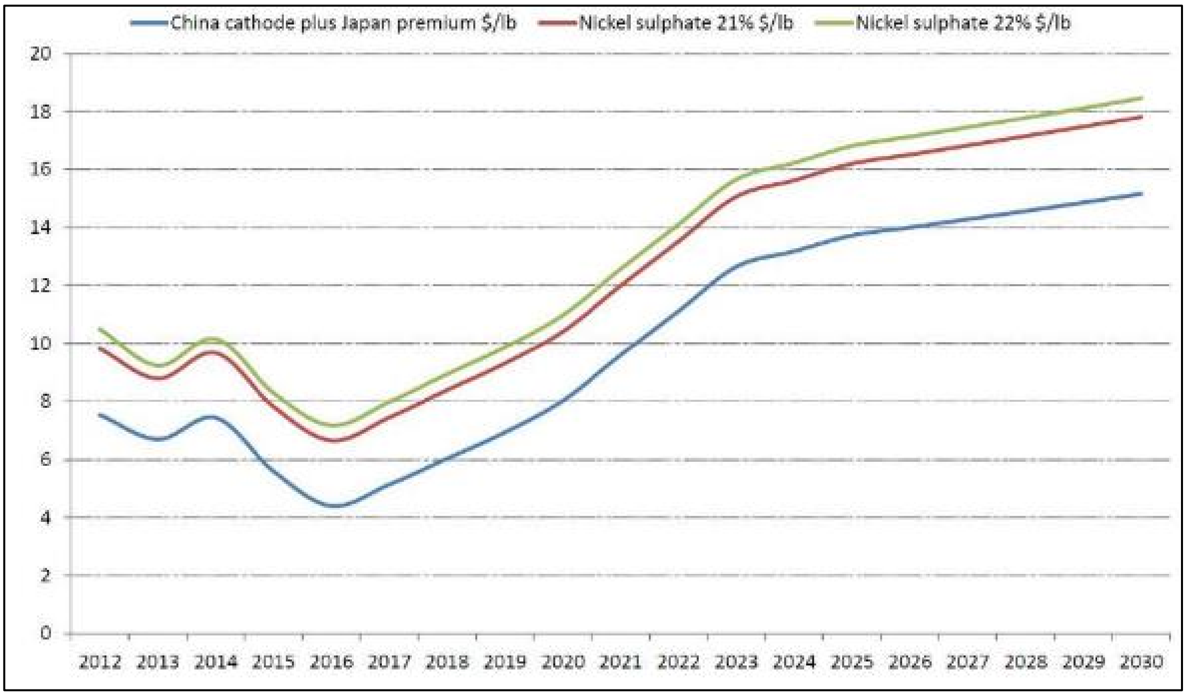



**Figure 19-7: Long-term nickel and nickel sulphate forecast [8]**

[8] CRU Report generated for Clean TeQ, 2016

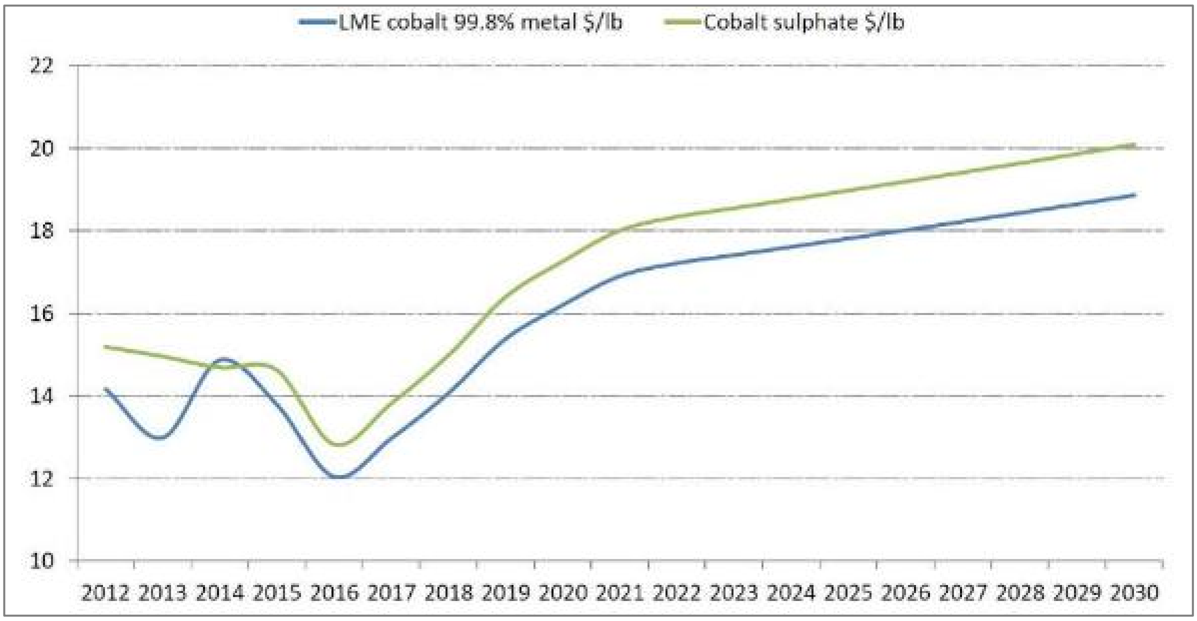



**Figure 19-8: Long-term cobalt and cobalt sulphate forecast [8]**

For the design purposes of this study, relatively conservative long-term pricing assumption of USD7.50/lb for nickel and USD12.00/lb for cobalt have been adopted, which includes assumptions around premia likely for these high purity products. It should be noted that no specific premia have been stated for the production of sulphate products, as it is not yet clear what the sustainability or quantum of premia may be over the longer-term. However, it is important to note that the premia represent potentially material upside to pricing outcomes. Further work is being undertaken with industry consultants to evaluate the long-term quantum (and sustainability) of potential premia for sulphate products over and above metal prices.

For economic analysis, metal prices are based on the long-term average consensus price forecasts from CIBC (as at Aug 25 2017) for a range of analysts. The long-term consensus nickel price was USD7.27/lb. However, previous consensus forecasts recently published by CIBC have been higher than this value, so a price of USD7.50/lb has been used to assess the economics of this Report. The long-term consensus cobalt price was USD14.85/lb and a USD14.00/lb cobalt price assumption has been used to assess the economics of this Report. As sulphate products are being provided and typically trade at a premium, assuming long term LME and LMB prices for nickel and cobalt respectively is considered to be conservative.

# 20 Environmental Studies, Permitting, and Social or Community Impact

## 20.1 Tenure and land access

The proposed mine operations area overlaps three pastoral properties. The Fifield State Forest and a parcel of Unoccupied Crown Land adjoin the north-eastern boundary of one of the pastoral properties. Two additional pastoral properties overlie the proposed limestone quarry.

### 20.1.1 Native title and freehold land

The Syerston project area does not lie within a registered Native Title claim area. Moreover, under the *Native Title Act 1993* (Cth), the valid grant of a freehold estate on or before 23 December 1996 effectively extinguishes Native Title. Virtually all of the land required for project implementation is freehold land (much of it owned by Scandium21), and thus there is minimal risk or any future project exposure to Native Title claims.

The company has signalled its intention to acquire additional freehold land, including properties known as "Slapdown" and "the Troffs" in order to secure its interests over the proposed mine operations area and limestone quarry (Figure 20-1). At the time of reporting, Clean TeQ have an agreement in place (documented and binding though not yet completed/ settled) to buy Slapdown and were negotiating an option agreement for Troffs which is also not completed.

Land acquisition and option agreements and easements are proposed to secure land required for the gas pipeline, rail siding and proposed bypass road around the town of Fifield. At the time of reporting, Clean TeQ had agreed terms to acquire the land which is the subject of the rail siding (documented and binding though not yet completed/settled).

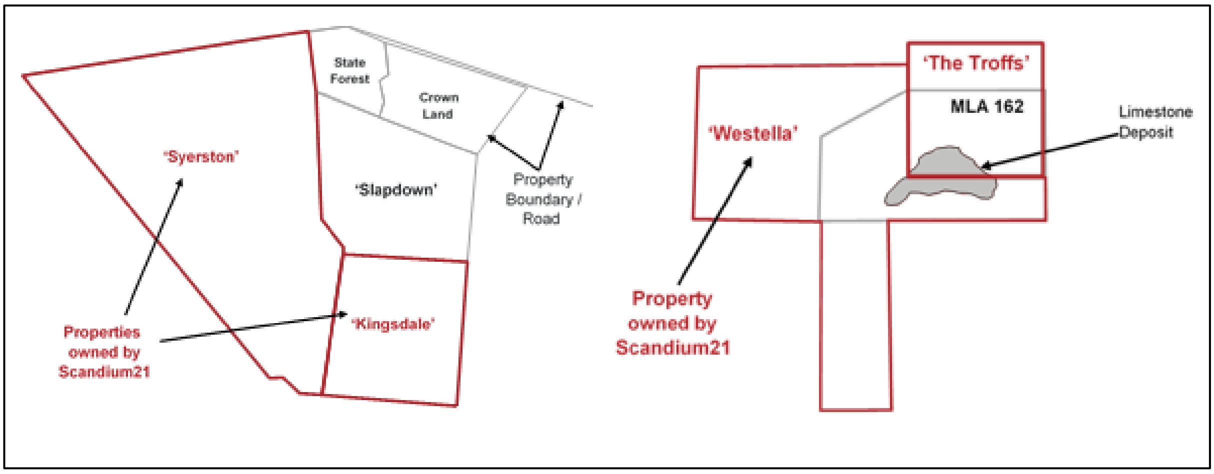



**Figure 20-1: Scandium21 landholdings and proposed land acquisitions**

Note: Main project area on left; limestone quarry on right.

### 20.1.2 Mining tenure

Scandium21 has been granted Exploration Licence 4573. The licence is due to expire on 16 August 2018, but can be renewed, subject to compliance with the provisions of the *Mining Act 1992* and to the licence conditions. The licence confers exclusive rights to prospect in the exploration area for Group 1 minerals, including rare earth minerals, nickel, cobalt and platinum.

The company has applied for granting of five mining leases over the proposed mine operations area and one lease (MLA 162) over the proposed limestone quarry. As at 31 July 2017, the NSW

Department of Planning and Environment (Resources & Energy) Minview database was showing the status of all mining lease applications as “pending”. Grant of mining tenure will be required before the commencement of mining.

**Table 20-1: Mining lease applications**

| Licence Application No. | Area | Application Date | Grant status (25/7/2017) | Interest |
|---|---|---|---|---|
| MLA 113 | 8 units | 10 August 1998 | Pending | 100% |
| MLA 132 | 200.00 ha | 20 September 1999 | Pending | 100% |
| MLA 141 | 137.5534 ha | 10 December 1999 | Pending | 100% |
| MLA 140 | 77.7845 ha | 10 December 1999 | Pending | 100% |
| MLA 139 | 421.0488 ha | 10 December 1999 | Pending | 100% |
| MLA 162 | 390.00 ha | 27 September 2000 | Pending | 100% |

## 20.2 Statutory approvals

### 20.2.1 Environmental Planning and Assessment Act 1979

All new mining projects (including expansions or modifications of existing projects) require development consent under the *Environmental Planning and Assessment Act 1979 (*EP&A Act) Projects with a capital investment exceeding AUD30 million are classified as “State Significant Development” and must be formally approved by the Minister for Planning (or his/her delegate).

The assessment process under the EP&A Act includes a structured system of technical assessments (Figure 20-2). The process also requires substantial stakeholder engagement. Project proponents are required to prepare and submit a detailed Environmental Impact Statement that addresses a range of environmental and other issues identified by the Secretary of the Department of Planning. The matters specified in the Secretary’s Environmental Assessment Requirements (SEAR) vary from project to project.

On 6 November 2000, a previous project owner, Black Range Minerals Pty Ltd, lodged a development application (DA) and an Environmental Impact Statement (EIS) with the NSW Department of Urban Affairs and Planning. Additional information relating to the management of groundwater, noise and air quality was submitted in January 2001, at the request of the EPA. Following a period of public comment and regulatory assessment, Development Consent DA 374-11-00 was issued under Part 4 of the EP&A Act in May 2001.

The Development Consent DA 374-11-00 has been modified on three occasions since it was issued:

- 2005 – to allow for the increase the run-of-mine (ROM) ore processing rate, limestone quarry extraction rate and adjustments to ore procession operations;
- 2006 – to allow for the reconfiguration of the water supply borefield; and
- 2016 - to allow mining and processing operations to initially focus on scandium oxide production and to approve adjustments to the processing operations to allow for the production of approximately 80 tpa of scandium oxide and up to 40,000 tpa of nickel and cobalt metal equivalents, as either sulphide or sulphate precipitate products. The project would then transition to the full-scale operation approved under earlier development consents.

Details of the most recent proposed project modifications were set out in a document entitled, “Syerston Project Scandium Oxide Modification Environmental Assessment” (Document No. 00740462, May 2016). The modified development application was approved on 12 May 2017, subject to a range of conditions set out in the “Notice of Modification”. The process for assessment and approval of State Significant Development makes provision for third party merit appeals and judicial reviews under some circumstances. Appeals are heard by the Land and Environment Court. SRK considers that stakeholder comment on the recent development approval have generally been supportive of the project and SRK considers it unlikely that third part appeals will arise.

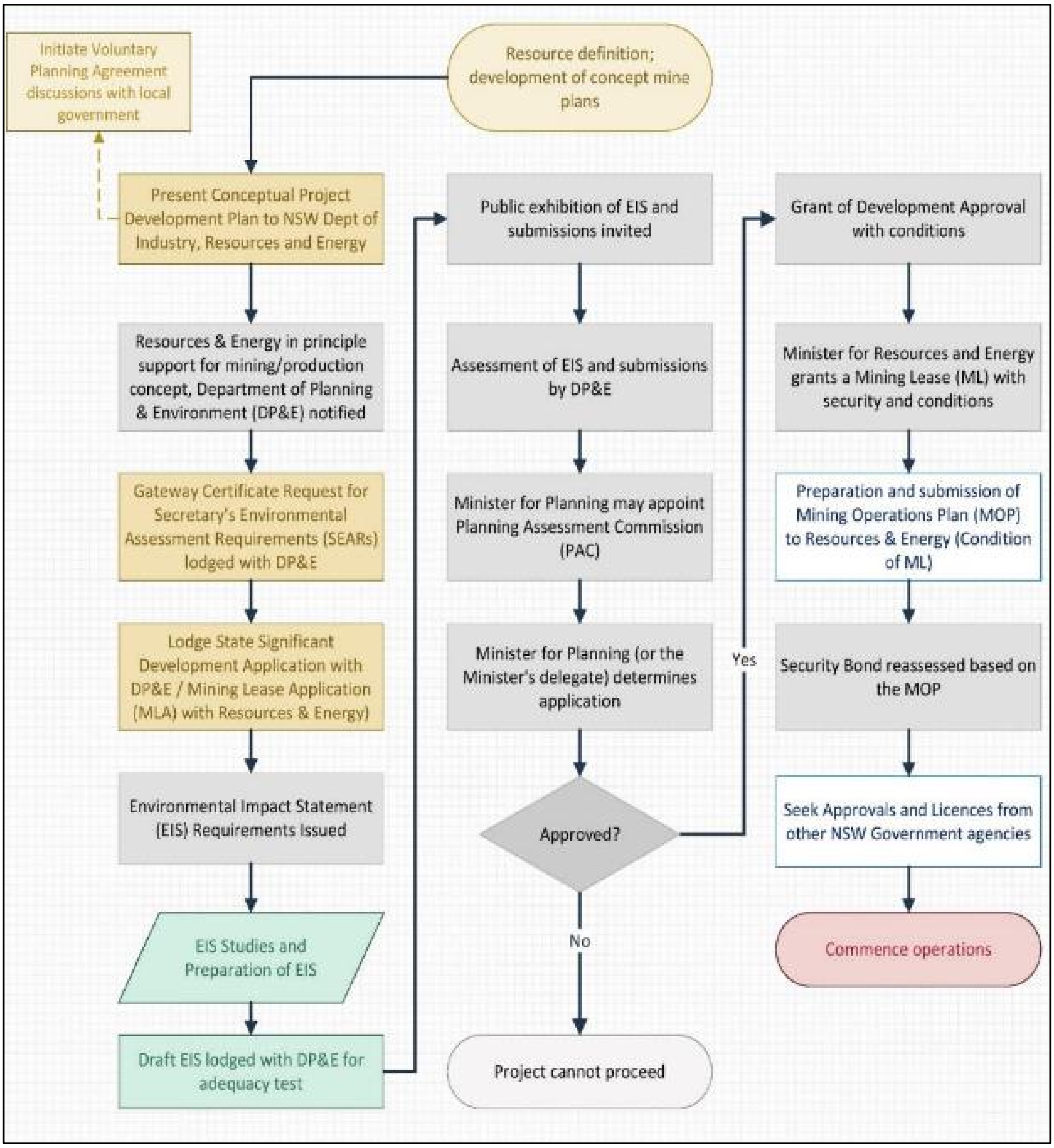



**Figure 20-2: Environmental impact assessment of state significant developments (simplified)**

The approval conditions in the most recent development approval include requirements for a range of pre-commencement actions that must be completed before the project may be implemented. These include preparation and implementation of a:

- Water Management Plan
- Noise Management Plan
- Air Quality Management Plan

- Blast Management Plan
- Biodiversity Management Plan
- Air Quality Management Plan
- Blast Management Plan
- Biodiversity Management Plan
- Heritage Management Plan
- Traffic Management Plan
- Rehabilitation Management Plan
- Revegetation Strategy
- Environmental Management Strategy.

Additionally, the project operator must complete a range of pre-construction and pre-commissioning hazard/safety studies and establish a community consultative committee prior to the start of operations.

The development approval recently granted for the Syerston project requires Scandium21 to implement "voluntary planning agreements" (VPAs) with each of the three local planning authorities within which the proposed developments occur. The terms of the VPA's are generally described in the development approval. They include an upfront contribution of AUD600,000 at start of construction (to be divided equally between the three shires), then ongoing annual contributions of approximately AUD566,000 (divided between the three shires) throughout the operational life of the project. Additionally, financial contributions are required for a range of road upgrades, safety audits and road maintenance activities.

### 20.2.2 Mining Act 1992

Environmental aspects of mineral exploration and mining (including mine rehabilitation and closure) in New South Wales are generally administered under the *Mining Act* 1992. Once a development consent has been issued under the *EP&A Act*, the project proponent is required to prepare and submit a Mine Operations Plan (including a mine rehabilitation plan) to the Department of Planning and Environment (Resources & Energy). Documentation required by Resources and Energy has a strong focus on mine rehabilitation. These documents has not been prepared at the time of reporting

Proponents are required provide an estimate of mine rehabilitation costs and to lodge a security deposit covering the full estimated cost of rehabilitation. Annual reporting on environmental performance (including rehabilitation progress) will be required as part of authorisations issued under the *Mining Act*.

At the time of reporting, no Mine Operations Plan or mine rehabilitation plan have been submitted to the NSW Resources Regulator. SRK has not reviewed the project's mine rehabilitation plan or any associated cost estimates.

### 20.2.3 Protection of the Environment Operations Act 1997

The *Protection of the Environment Operations Act* (POEO Act) is the instrument through which certain specified activities are regulated by the Environment Protection Authority (EPA). Activities listed in Schedule 1 of the POEO Act are administered by means of environmental protection licences issued to owners or operators of the premises on which the activities occur. The Syerston project would require a licence authorising mining and mineral processing, as well as some other regulated activities (potentially including road construction, waste disposal, sewage treatment, quarrying of borrow material, hazardous chemical storage and the like). SRK understands that Scandium21 has not yet lodged any applications for a POEO licence (as at 25 July 2017).

### 20.2.4 Environment Protection and Biodiversity Conservation Act 1999

In some circumstances, projects must be assessed and approved under both State and Commonwealth environmental legislation. Black Range Minerals Limited referred the Syerston project to the Commonwealth by for possible assessment under the *EPBC Act* in January 2001. However, the Commonwealth determined that the proposed mining activities did not constitute a “controlled action” under the Act. Accordingly, no federal assessment or approval is required.

### 20.2.5 Other approvals

A number of other consents may be required prior to the commencement of project implementation. The chief among these are consents under the *Water Management Act* 2000, as discussed in Section 18.4 of this report. Approvals from the NSW Dams Safety Committee would be required for tailings storage facilities and any other significant water impoundments.

Scandium21 currently holds a water allocation (Water Access Licence 32068) which enables abstraction of up to 3.154 GLpa of groundwater from the Upper Lachlan Alluvial Zone 5 Management Zone (Upper Lachlan Alluvial Groundwater Source). Although this amount of water would most likely be sufficient for the initial production phase of the modified project (processing up to 100,000 tpa of ore), it would fall well short of the water required to operate the project at the full 2.5 million tpa throughput approved under the current approval.

Documentation provided for the previous and current development approvals provides little information as to how the full project water demand will be satisfied. The conditions attached to the most recent environmental consent seek to address this uncertainty by imposing a number of pre-commencement obligations including (but not limited to) requirements to:

- prepare a Water Management Plan for the development in consultation with DPI Water and the EPA. The plan must include a water balance with details of water sources and water supply contingency measures. and
- provide compensatory water supplies to owners or leaseholders whose basic water user rights are directly impacted by the project.

SRK notes that the current approved volume of water sourced from borefields is not sufficient to meet the Project’s entire raw demand and was to be supplemented with water direct from the Lachlan River. At the time of writing this report, approvals were not obtained for this supplementary water source. While the current project is reducing water demand, it is likely that supplementary water will be required.

Clean TeQ plan to finalise the water balance as well as assessing options to secure rights to any additional water requirement as part of the FS.

## 20.3 Potential environmental constraints

The key permitting constraint identified by SRK for the Syerston project relates to consents required to source and use water (whether surface water or groundwater) to satisfy operational water requirements. The primary development consent required for project implementation was granted in May 2017, subject to a range of conditions, including completion of an array of pre-commencement studies and preparation of management plans. The pre-commencement conditions relating to project water supply and water management will require a significant amount of technical work, stakeholder consultation and administrative effort.

A number of important secondary approvals must be granted before the project can be implemented and SRK has seen no evidence that applications for the key secondary consents required under the Mining Act and the Protection of the Environment Operations Act have yet been lodged with the relevant authorities.

No significant issues related to Native Title, Aboriginal heritage, biodiversity protection or pollution control have been identified that would materially constrain project permitting or implementation.

## 20.4 Environmental aspects of project cost

The development consent recently granted for the Syerston project commits the project operator to ongoing annual contributions of approximately AUD600,000 to various local government initiatives. Other environmental costs likely to arise include:

- Cost of developing and Implementing an environmental management system (including development of a comprehensive suite of environmental management plans);
- Pollution control design components (TSF lining and cover system, bunding and drainage control in operational areas, control of dust and other emissions to air);
- Ongoing environmental monitoring and compliance activities;
- Mine rehabilitation works;
- Revegetation costs; and
- Environmental security bonds.

Project documentation available to SRK did not include any analysis of greenhouse gas emissions for the project, these may be substantial for processes involving limestone neutralisation of acidic process streams. Given the duration of the project, and the evolving nature of greenhouse policies, future project cost reviews should consider the possible cost implications of the project’s CO2 emissions.

## 20.5 Summary and conclusions

The primary environmental consent required for project implementation has been granted. Based on a review of decisions for comparable recent developments in the general locality.

With the possible exception of water allocation permits, SRK has no reason to expect that the secondary approvals required for the project would be refused.

# 21 Capital and Operating Costs

## 21.1 Capital costs

A capital cost estimate for the design and construction of a 2.5 Mtpa HPAL processing plant incorporating RIP processing was developed in-house by Clean TeQ at a PFS level of accuracy. It was based on the earlier September 2005 Feasibility Study capital cost estimate completed for the Project by SNC-Lavalin (Australia) and JGC Corporation (Japan). These costs were escalated and further modifications made, largely the inclusion of a RIP circuit, road upgrade allowance and adjustments to other areas based on the associated changes with the modified flowsheet.

SNC-Lavalin were engaged to undertake a review of the original costs (2016) and the price escalation since the original work (2005) was undertaken. This focused on Artisan Labour rates, structural steel pricing and mechanical equipment pricing a further review also considered the factors for the following disciplines:

- Civil/ earthworks
- Concrete
- Structural steel
- Buildings
- Mechanical equipment
- Piping
- Electrical
- Instrumentation
- Indirect costs.

SNC-Lavalin's review concluded that there is likely to have been an overall reduction in pricing of between 3 and 5% due to a significant softening of the construction and resource sector over that period. The review that the Clean TeQ estimate was appropriate and practical representation of the anticipated capital costs and that based on the original estimate was at a feasibility study level of accuracy, the Clean TeQ revision was likely to fall within a +/-20% accuracy band.

A Feasibility Study has now been initiated to develop the full plant costs from first principals but as an interim measure, to further improve the confidence in the capital costs, Clean TeQ has developed the RIP plant costs from first principals and engaged Simulus Engineers to develop the refinery capital costs to better reflect the downstream processing circuits, i.e. post Ni/Co RIP, to a PFS level of accuracy. These costs have been built up using mechanical equipment costs, material take-offs for civil and structural steel work, piping and valves, electrical load list from the mechanical equipment, with other engineering discipline costs such as instrumentation and control factored.

Indirect costs include EPCM costs, owner's costs, first fills and spares, freight, owners' costs and additional construction indirects. Owner's costs include; project management, insurance, land acquisition, commissioning, geology, some head office overheads, independent engineer costs and an Owner's input to the engineering and procurement, environmental management and pre-production costs.

Items not included in the estimated contractor rate that will be required or supplied by construction management or Owner were as follows: EPCM Contractor construction facilities, including offices, warehouse, crib rooms, hardstands, water, temporary power and communications infrastructure; Constructor facilities including water, sewerage, temporary power, communications infrastructure; Temporary fencing; Material testing; Temporary roads and laydown area; Concrete batch plant;

Construction camp; Diesel supply; Air terminal to camp transport; and Camp to job site transport.

The updated RIP and refinery capital cost estimates have then been consolidated with the direct front-end costs for beneficiation, HPAL and partial neutralisation which remain largely unchanged in design as well as the indirect costs. The contingency of 10% has been increased to 15%, which is more appropriate at this level of study.

The capital costs are AUD1,045M, based on the updated PFS cost estimates. Costs were based on an EPCM basis, incorporating contingency and are considered to be within the +/-25% accuracy band expected of a PFS level of study accuracy. The estimate base date is Q3 2017.

The estimate includes all site preparations and bulk earthworks, process plant and associated utilities including acid plant, power station, remote borefield and limestone milling, first fills and capital spares, site buildings, TSF and evaporation ponds, water supply and distribution systems, site access road and external road upgrades, site buildings, diesel storage, plant control system, IT and communications, security, owner's costs, temporary construction logistics, owners' costs and other miscellaneous requirements.

A summary of capital costs is presented below in Table 21-1.

**Table 21-1: 2.5 Mtpa capital cost estimate**

| Plant Area | Cost | |
|---|---|---|
| **Currency** | **USDM** | **AUDM** |
| Mining | 14 | 18 |
| Site Preparation | 11 | 15 |
| Process Plant | 210 | 280 |
| Process Utilities | 85 | 113 |
| Services | 110 | 146 |
| Infrastructure | 45 | 60 |
| **Total Directs** | **474** | **632** |
| Indirects, including EPCM | 105 | 140 |
| Owners Costs, including spares and first fills | 103 | 137 |
| **Capital Cost, excluding contingency** | **682** | **909** |
| Contingency (15% of Directs/Indirects) | 102 | 136 |
| **Total Capital Cost Estimate** | **784** | **1,045** |

Note: USD:AUD = 0.75.

The base case capital cost estimate does not include a scandium recovery and purification circuit. No scandium revenues have not been assumed in the base case financial model.

### 21.1.1 Exclusions

The capital cost estimate excludes a number of supporting infrastructure costs. These are expected to be provided on a Build, Own and Operate (BOO) basis or direct supply cost, and are not incorporated into the capital cost estimate. The exclusions include:

- Mining operation including fleet, buildings and workshops
- Limestone from owner's quarry based on a per tonne supply cost
- Supply of all reagents, including sulphur port and rail facilities
- Liquid nitrogen plant

- Mobile plant and equipment required for operations
- Site mobile vehicles plant
- Natural gas pipeline.

Re-engagement and initial discussions have been held with a number of the key third providers for their supply however the technical details and commercial terms have not been finalised, largely due to the level of study currently being undertaken. In most cases this leverages the engineering work currently undertaken during the 2005 Feasibility Study. Firmer pricing will be developed during the next phase of study during more formalised negotiations.

Other exclusions include:

- Financing costs
- Foreign exchange variations
- Schedule delays.

### 21.1.2 Currency

The estimate is expressed in Australian Dollars and US Dollars unless noted otherwise, assuming a USD0.75: AUD1.00 exchange rate for the bulk of the costings but with some undertaken at USD0.76: AUD1.00 for some of the latest cost updates. Other currency exchange rates were based at the time of estimation as is conventional practice.

### 21.1.3 Escalation

Capital costs for the front end of the processing plant and some utilities and supporting infrastructure, specifically ore preparation and milling, HPAL, tailings neutralisation and the acid plant, were estimated using the capital estimates previously developed in the 2005 Feasibility Study Update are similar to costs today, based on relative structural steel and labour rates in an Australian context.

SNC-Lavalin completed a high-level review of the 2005 capital cost estimates in August 2016, to determine whether any material variations (taking account of cost escalation since 2005) needed to be included. This work focused on local labour rates, steel and other construction material costs and (for operating costs) current market prices for key raw material inputs for processing operations (sulphur, natural gas, etc).

While raw material prices peaked in 2006 - 2008, particularly titanium and steel, prices have now returned to similar levels used during the 2005 Feasibility Study Update. A recent study into labour rates for the Company’s scandium Feasibility Study has shown that average labour rates have returned to the levels seen during the 2004-2005 period. This was confirmed by the SNC-Lavalin review.

### 21.1.4 Working, deferred and sustaining capital

The capital estimate does not include an allowance for working, deferred and sustaining capital. However, an allowance has been made in the financial modelling (see Section 22). Sustaining capital was assumed to be a flat 1.25% of Direct Costs. AUD5M of surface water purchases to supplement the borefield was also assumed over the first 4 years starting in Year 1. The second TSF cell construction cost of AUD4.48M, for tailings dam north, is assumed to occur in Year 3.

### 21.1.5 Taxes and duties

Goods and Services Tax (GST) is not included in the estimate.

### 21.1.6 Owner's contingency

Owners Contingency has been included in the estimate as a cost of 15% of the total cost. This contingency has been increased from the original 8% provided for in the 2005 Feasibility Study Update detailed estimate.

### 21.1.7 Estimate accuracy

The Accuracy of the capital estimate is ±25%, within the accepted range of a PFS.

## 21.2 Operating costs

Clean TeQ developed operating cost estimates to support the Syerston Nickel Cobalt PFS, based largely on the work undertaken during the 2005 FS. They were originally built up from first principles. Input values updated to reflect new contract mining, reagent, utility prices and labour costs. The stated accuracy of the estimate is reported to be within ±20% as at Q4, 2016. The claimed accuracy range is consistent with PFS guidelines.

The annual operating costs for the project over the first 20 years of operation are summarised below in Table 21-2. It is based on Years 3 to 20 as tonnage is still ramping up in Years 1 and 2. The operating cost includes mining, processing, product production and logistics to offtake customers. An exchange rate of USD0.75:1.00AUD was assumed.

**Table 21-2: Operating cost summary (Years 3 - 20), excluding transportation**

| Cost Centre | Total cost (AUDM) | Total cost (%) | Unit cost (AUD/t ore) | Unit cost (USD/lb Ni) | Unit cost (USD/lb Ni) after Co credits |
|---|---|---|---|---|---|
| Mining | 39.6 | 20.7 | 15.86 | 0.72 | |
| Ore | 142.3 | 74.2 | 56.90 | 2.58 | |
| Utilities | 0.1 | 0.1 | 0.03 | 0.00 | |
| Services & Infrastructure | 0.8 | 0.5 | 0.34 | 0.02 | |
| Finance & Admin | 8.9 | 4.6 | 3.56 | 0.16 | |
| **Total** | **191.7** | **100** | **76.69** | **3.48** | **1.42** |

All costs are estimates, and are subject to the constraints and qualifications described in the following sections.

### 21.2.1 Exclusions

Operating costs exclude: Realisation costs which includes sea freight transport and marketing costs; Working capital movements which includes nickel and cobalt receivables, creditors and inventory costs; Additions or deletions associated with further testwork; Owners costs (Head Office); Royalty payments; Exploration costs for areas other than current defined orebody; Escalation; Depreciation and accounting effects; Exchange rate fluctuations; Corporate Tax; and Financing. These have been factored into the base case financial model.

### 21.2.2 General sales tax

To the extent that production is exported and therefore not consumed within Australia, production will be GST-free, meaning that no GST will be charged and collected from customers on the sale of the product.

### 21.2.3 Personnel

Annual salary rates from the 2005 FSU were reviewed by Clean TeQ and determined to be in line with current market expectations. A total of 35% on-costs were assumed for the labour cost estimate.

Mobile equipment is based on the light vehicle requirements within the process plant, Company mining personnel, environmental, engineering and administration. This does not include maintenance vehicles. The total estimated cost for mobile equipment is distributed to the Mining, Ore Leach and Finance & Administration cost centres. The distribution is 14%, 68% and 18% respectively and is related to the estimated distribution of personnel between these cost centres and their vehicular requirements.

### 21.2.4 Mining

The basis for the mining contract is the mine schedule, which is discussed in detail in Section 16. Indicative contract mining costs were obtained in order to update the inputs into the PFS operating cost estimate. A summary of these costs is presented in Table 21-3.

**Table 21-3: Open pit mining operating costs**

| Items | Mining Operating Cost (AUD/bcm ore mined) | Mining Operating Cost (AUD/bcm total material mined) |
|---|---|---|
| Mobilisation & Demobilisation | 0.01 | 0.005 |
| Clear & Grub | 0.09 | 0.03 |
| Topsoil | 0.09 | 0.03 |
| Haul Road | 0.04 | 0.01 |
| Load & Haul | 15.07 | 4.58 |
| Drill & Blast | 0.00 | 0.00 |
| Pumping (dewatering) | 0.04 | 0.01 |
| Rehandle | 0.44 | 0.13 |
| ROM Pad | 0.07 | 0.02 |
| Dayworks | 1.56 | 0.47 |
| Grade Control | 2.33 | 0.71 |
| Other | 0.01 | 0.00 |
| Rehabilitation | 0.01 | 0.00 |
| Total | 19.77 | 6.00 |

### 21.2.5 Processing

The Process Plant includes the following cost centres: Ore Leach; RIP, Refinery; Utilities; Services; and Infrastructure. The majority of the estimated Utilities and Services costs were distributed into the Ore Leach area. All of the estimated Infrastructure costs were distributed to these main areas.

Processing costs were initially built up from the 2005 plant mass and energy balance, with updates made for the RIP and refinery areas. This balance also forms the basis of the mechanical equipment list and therefore the associated electrical load list. Power, steam, acid and other reagent consumptions are provided by the mass and energy balance, with input prices obtained for each.

The requirements for high temperature leaching acid consumption were estimated based on the ore composition in each period. This relationship was developed from variability leach metallurgical testwork. Natural gas costs are estimated on the requirement for power and process steam.

Reagent pricing was based on specifications and quantity estimates. Written quotations for reagent supply costs were sourced for the 2016 Scandium Feasibility were also used for this nickel and cobalt PFS. The combination of sulphur, natural gas and limestone represents a significant percentage of the total operating cost.

Maintenance costs are factored from the direct capital costs. Maintenance consumables, such as filter cloths, cyclone liners etc., estimated by calculation, industry normal consumption rates, or vendor supplied consumption rates, depending on the item, were assumed to be the same for this Study. Maintenance Consumables represented an estimated 1% of the total operating cost. Where maintenance consumable prices were not available the cost was estimated based on the area Capex and included in the operating cost model as a Maintenance Spares cost. Maintenance Spares were estimated as a percentage of the total installed direct capital cost. They do not include items already accounted for separately in operating consumables.

To establish the cost per tonne of feed and metal production, the mine schedule has been used as the basis for estimated autoclave feed tonnage and grade. In the first two years, the ramp-up schedule throughput, availability and recovery values have been used.

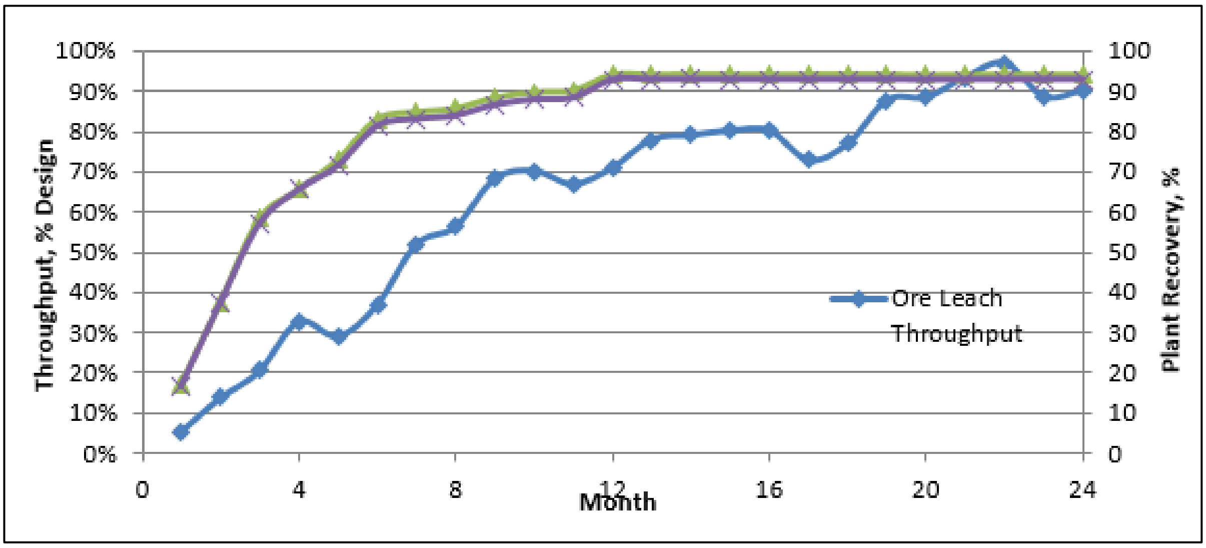



**Figure 21-1: Example ramp up schedule**

## 21.2.6 General and administrative (G&A)

General and administrative cost estimates have been undertaken for the Syerston Project. They are based on the 2005 FS estimate, but have been updated for the August 2017 Syerston Scandium Project FS. The G&A costs comprise: general expenses; access road maintenance; light vehicle costs; personnel costs; and site buildings. Personnel and light vehicle costs have been discussed in previous sections. These costs have been assumed to remain the same and equal for both operations considered.

## 21.2.7 Estimate accuracy

The costs have been initially reviewed or factored from the original 2005 Feasibility Study Estimate which was undertaken to an accuracy of -5% +15%, or in the case of the RIP circuits based on Clean TeQ's in-house database based on the recent Scandium Feasibility Study completed. Simulus Engineers has also updated the post RIP refinery operating costs to improve the operating cost estimate confidence. The accuracy of the operating cost estimate is ±25%, within the accepted range of a PFS level of study.

## 21.2.8 Contingency

There has been no allowance made in the operating cost estimate for operating cost contingency. This is normal practice for operating cost estimates.

# 22 Economic Analysis

The valuation of a mine producing nickel sulphate and cobalt sulphate has been untaken via an assessment of the discounted cash flow for the life of the Project. The valuation model is based on the following development plan:

- Mining via contract mining operations of various pits to a ROM pad;
- Processing the ore through the Processing Plant at a production rate of 2.5 Mtpa;
- Recovery of nickel and cobalt and production of separate hydrated sulphate products;
- Recovery of scandium and production of scandium oxide (99.9% purity) - optional; and
- Placement of treated tailings into a newly constructed tailings dam.

A cash flow model was based on inputs from the technical model and the factored engineering estimate completed by SNC-Lavalin.

## 22.1 Inputs and assumptions

The valuation assumes a 20-year project life, although the resource is sufficient to support an operation beyond this valuation period. The model is developed based on an estimate of the real costs of operations and therefore does not include any allowances for inflation on either costs or revenues. This provides a more accurate estimate for the long-term average costs and revenues for the project. Table 22-1 summarises the key inputs and assumptions:

**Table 22-1: Syerston valuation model key inputs**

| Parameter | Unit | Value |
|---|---|---|
| Autoclave Throughput | tpa | 2,500,000 |
| Average Strip Ratio | 1: | 0.80 |
| Average Production (Years 3 - 20) | | |
| Nickel Sulphate ($NiSO_4.6H_2O$) | tpa | 85,135 |
| Cobalt Sulphate ($CoSO_4.7H_2O$) | tpa | 15,343 |
| Recovery (20-year average) | | |
| Ni | % | 94.2 |
| Co | % | 93.0 |
| Life of Mine (initial) | years | 20 |
| Nickel Long Term Price | USD/lb | 7.50 |
| Cobalt Long Term Price | USD/lb | 14.00 |
| Exchange Rate (AUD: USD) – Life of Mine | 1 : n | 0.75 |
| Discount Rate | % | 8 |
| Tax Rate | % | 27.5 |
| Royalties | | |
| NSW Government | % | 4 |
| Ivanhoe Mines (after Govt royalty) | % | 2.5 |
| Depreciation | % | 20% declining balance |
| Sustaining Capital (all years) | % of Directs | 1.25 |

For economic analysis, metal prices are based on the long-term average consensus price forecasts from CIBC (as at Aug 25 2017) for a range of analysts. The long-term consensus nickel price was USD7.27/lb. However, previous consensus forecasts recently published by CIBC have been higher than this value, so a price of USD7.50/lb has been used to assess the economics of this Report. The long-term consensus cobalt price was USD14.85/lb and a USD14.00/lb cobalt price assumption has been used to assess the economics of this Report. As sulphate products are being provided and typically trade at a premium, assuming long term LME and LMB prices for nickel and cobalt respectively is considered to be conservative.

### Exchange rate

An average long term exchange rate of 0.75USD:1.00AUD has been used for the valuation model. An analysis of the effect of the exchange rate on project valuation has been provided in Section 22.2.

### Taxes

A 27.5% company tax was applied for all years.

The Australian Government has an R&D Tax Concession program which allows a 43.5% refund on eligible R&D expenditure. The majority of the eligible expenditure would be in Year 1 during the commissioning and ramp up period. However, no R&D tax concessions have been considered in this valuation.

Goods and Services Tax (GST) applies to most sales items. In the case of the Project this would be on all consumables, reagents and utilities purchased in Australia. Typically, a portion of this tax is recoverable out of corporate taxes or is directly reimbursed. However, this valuation makes no allowance for GST and therefore any recovery of GST has not been considered.

All product sales are assumed to be to overseas customers. Export sales of nickel and cobalt sulphate and scandium oxide are exempt from GST and therefore not applied to the sale of scandium.

### Royalties

A 4% NSW government mineral royalty has been applied over the life of mine. The calculation is as follows:

Royalty = (Revenue - Processing Cost – (33% x Site Admin Cost) – Allowable Depreciation Deduction - Transport Costs) x 4%

Additionally, a 2.5% royalty payable to Ivanhoe Mines has been applied over the life of the mine. The calculation for this royalty is as follows:

Ivanhoe Mines Royalty = (Revenue – NSW Royalty) x 2.5%

### Depreciation

Depreciation was calculated using a diminishing value methodology. It was assumed that the project depreciation is the same as the accounting depreciation and in real terms (i.e. no inflation considered) for the purposes of the study.

## 22.1.1 Discounted cash flow analysis

Table 22-2 summarises the discounted cash flow valuation for the Project for the first 20 years of operation, based on the assumptions in Table 22-1 and the production schedules and grades outlined for each flow sheet in Section 16.

**Table 22-2: Discounted cash flow valuation**

| Parameter | Unit | Value |
|---|---|---|
| NPV (post tax) | A$M | 996.7 |
| | USDM | 747.5 |
| IRR (post tax) | % | 20.7 |

It is assumed that the separate nickel sulphate hexahydrate and cobalt sulphate heptahydrate products are purchased outright by a third party or off-take partner. While the sulphate product typically trades at a premium, the long-term LME price for both nickel and cobalt have been assumed. Accordingly, there is potential upside in pricing should nickel and cobalt sulphate premia continue to apply over the life of the mine.

## 22.2 Sensitivity analysis

In order to determine the robustness of the project and highlight the key variables impacting the economics, a sensitivity analysis was performed. This was performed by changing one variable at a time, holding all other variables constant. Key process and economic variables were selected with ranges selected which represented both reasonable upside and downside.

### 22.2.1 Net present value analysis

The inputs were individually varied to determine their effect on the overall NPV (post tax) of the project. The results of this analysis are shown in Table 22-3. The spider analyses indicate that the parameters which have the greatest incremental impact are feed grade, payability and nickel price. Payability is defined as the premium or penalty imposed on the products produced at the site. As high quality sulphate products are produced, these have the potential to attract a premium of 10-25%.

**Table 22-3: NPV analysis (post tax)**

| $NPV_8$ (USDM) | +20% | +10% | Base | -10% | -20% |
|---|---|---|---|---|---|
| Autoclave Feed Grade | 1,185.0 | 966.5 | 747.5 | 528.5 | 308.5 |
| Capital costs | 636.4 | 691.9 | 747.5 | 803.1 | 858.7 |
| Operating costs | 562.5 | 655.0 | 747.5 | 840.0 | 932.5 |
| A$/USD | 576.8 | 662.2 | 747.5 | 832.9 | 918.2 |
| Payability | 1,185.0 | 966.5 | 747.5 | 528.5 | 308.5 |
| Nickel Price | 1,074.6 | 911.1 | 747.5 | 583.9 | 420.1 |
| Cobalt Price | 858.4 | 802.9 | 747.5 | 692.1 | 636.7 |

### 22.2.2 Internal rate of return analysis

Inputs were individually varied to determine their impact, Table 22-4, on the overall post-tax IRR of the Project. Similar to NPV, IRR is affected by the feed grade, nickel price and payability.

**Table 22-4: IRR analysis (post tax)**

| IRR (%) | +20% | +10% | Base | -10% | -20% |
|---|---|---|---|---|---|
| Autoclave Feed Grade | 27.3 | 24.3 | 21.0 | 17.6 | 13.9 |
| Capital costs | 17.8 | 19.3 | 21.0 | 23.1 | 25.5 |
| Operating costs | 18.1 | 19.6 | 21.0 | 22.5 | 24.0 |
| A$/USD | 18.3 | 19.7 | 21.0 | 22.4 | 23.8 |
| Payability | 27.3 | 24.3 | 21.0 | 17.6 | 13.9 |
| Nickel Price | 25.7 | 23.4 | 21.0 | 18.6 | 15.9 |
| Cobalt Price | 22.8 | 21.9 | 21.0 | 20.2 | 19.3 |

# 23 Adjacent Properties

There are no resource projects/ properties immediately adjacent to the Project that are materials to the findings of this Report. There are several large mining projects in the region as shown in Figure 23-1.

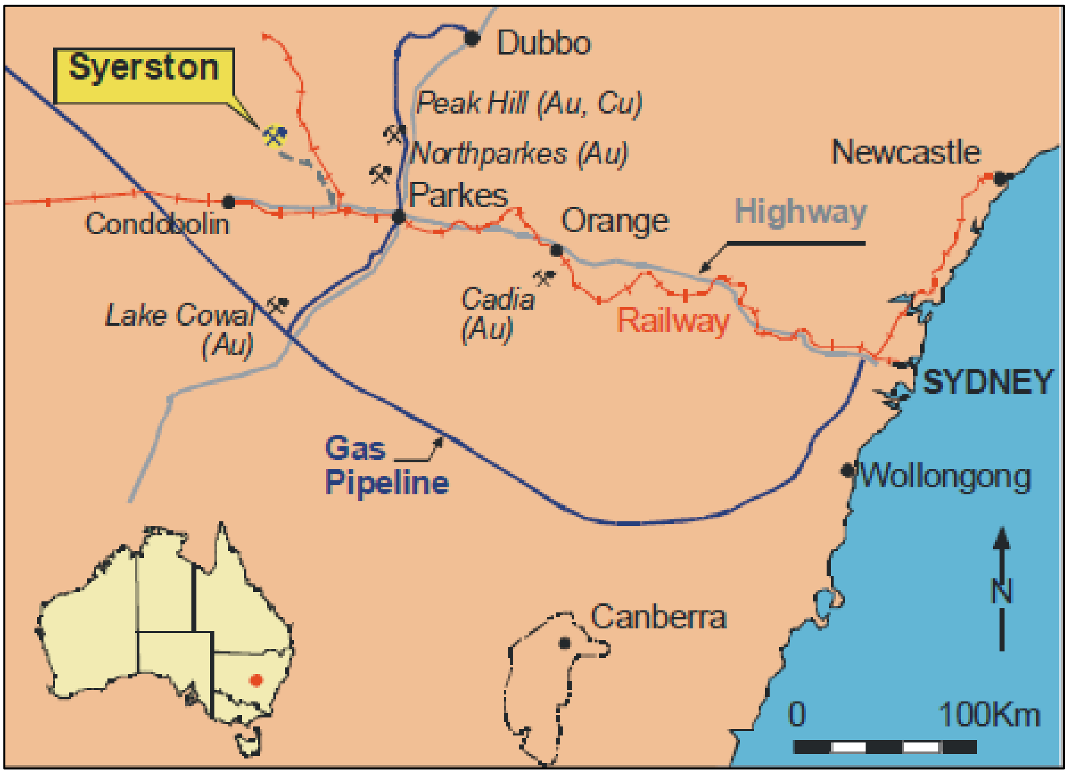



**Figure 23-1: Other resource projects in the regional vicinity**

# 24 Other Relevant Data and Information

This Report focuses on the extraction of nickel and cobalt and does not focus on the extraction of Scandium. SRK is aware that Clean TeQ are progressing additional work to assess the technical and economic feasibility of the production of Scandium.

Where appropriate SRK makes reference to Scandium for context but no value has been attributed to Scandium in this assessment.

Clean TeQ completed a scoping study in May 2015 focusing on the recovery of scandium from geologically distinct high grade zones adjacent to and exclusive of the nickel/cobalt deposit (Figure 24-1). In July to October, 2015, Clean TeQ undertook a large-scale pilot plant program on approximately 12 tonnes of high-grade scandium ore from site. Subsequent to this work, detailed metallurgical testwork on the scandium process was carried out.

A Feasibility Study for the scandium project was completed on 30 August, 2016. Although, processing just 64 000 tpa, the scale of production of this study produced a quantity of product in excess of three times the current word wide annual demand for scandium requiring significant future development of the Scandium market to achieve the stated outcomes of the study.

The development of the scandium project has been carried out to allow for the development of the separate and much larger nickel and cobalt operation at a later date. As the high-grade scandium and nickel/cobalt zones are on separate areas these projects may be developed independently. In the case of joint development, the projects will be able to utilise common infrastructure and services, however it is the Company’s view to develop only one Project at this time.

In a scandium context, the development of HPAL for the extraction of scandium is widely accepted. Metallica Minerals completed a Pre-Feasibility study based on HPAL for scandium extraction. This included extensive metallurgical testwork. Both the Owendale (Platina), Flemington (Australian Mines) and the Nyngan (Scandium International Mining Corporation) projects have completed testwork programs and studies, including metallurgical testwork validation of HPAL for scandium extraction, with recoveries similar to that of nickel and cobalt.

The metallurgical testwork completed in the previous two feasibility studies on Syerston typically followed scandium, as well as nickel and cobalt. All historical testwork confirms that scandium extraction using HPAL ranges from 80% - 90% and higher.

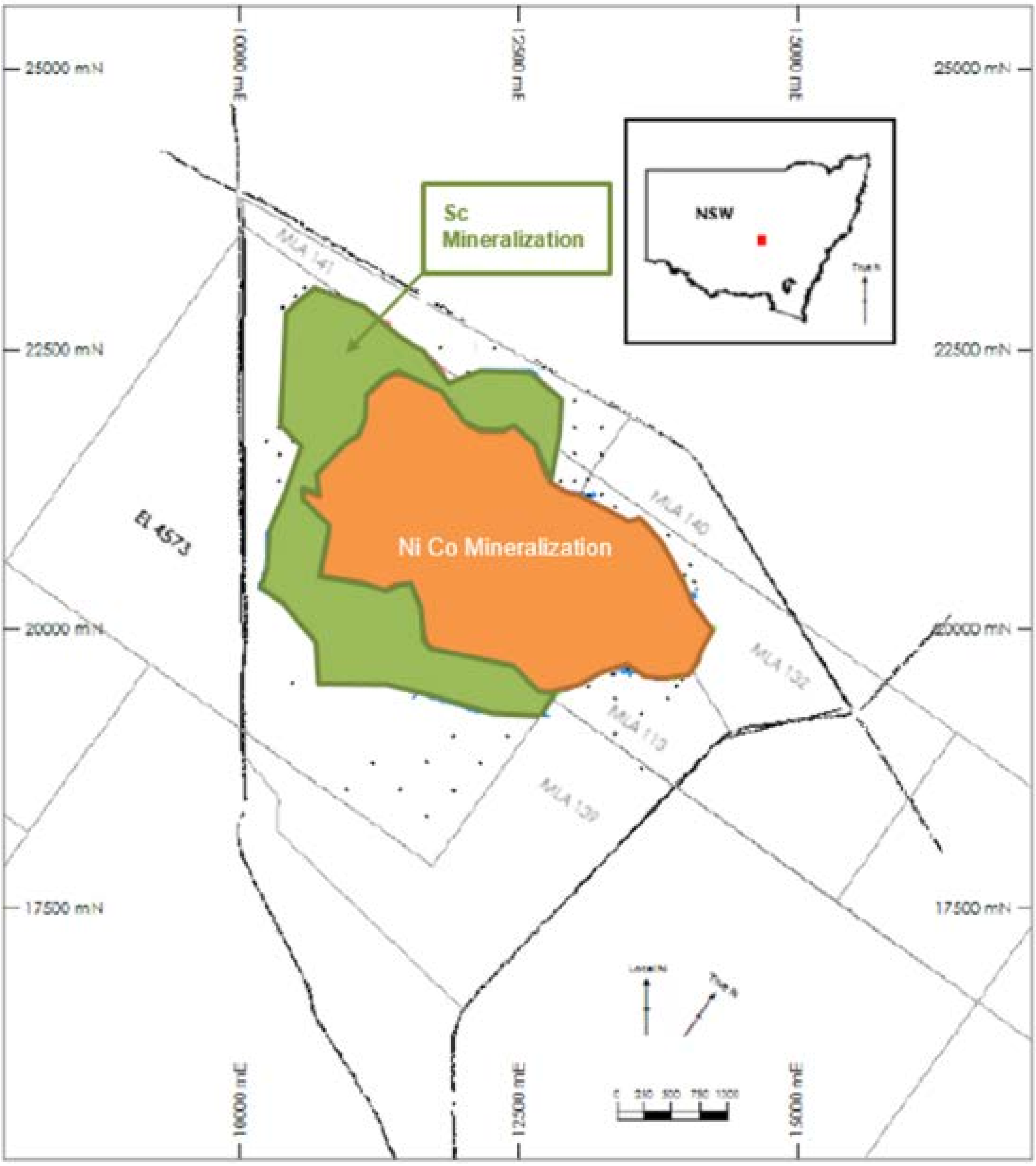



**Figure 24-1: Scandium mineralisation location in relation to nickel cobalt mineralisation**

# 25 Interpretation and Conclusions

Results of SRK's review of the 2016 Pre-Feasibility Study and supporting documentation demonstrate that the Syerston Ni/Co project warrants further studies due to its positive economics.

It is the conclusion of the QPs that the 2016 PFS summarises in this Technical Report contains adequate detail and information to support a pre-feasibility level analysis. Standard industry practices, equipment and design methods were used in this Pre-Feasibility Study and except for those outlined in this section, the report authors are unaware of any unusual or significant risks or uncertainties that would affect project reliability or confidence based on the data and information made available.

For these reasons, the path going forward must continue to focus on updating the mining operating costs, confirming water requirements and securing water sources, finalising land agreements.

Risk is present in any development project. Pre-feasibility engineering formulates design and engineering solutions to reduce that risk common to every mining project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, environmental and social impact, schedule and cost overruns, and labour sourcing. SRK is of the opinion that these risks have been clearly identified and mitigation measures have been considered.

Potential risks associated with the Project are summarised in Table 25-1.

**Table 25-1: Project risks and opportunities**

| Risk/ Opportunity | Summary |
|---|---|
| Technology | • The HPAL process is now in its fourth generation, with plants successfully operated since the 1960s. Many of the learnings of previous operations were incorporated in the 2005 FS. Since that time, a number of new operations have commenced. Suitably qualified engineers will be used for any revised design work for the HPAL system. Similarly, RIP technology has successfully been implemented since the 1950s on over 30 full-scale operations to recover a range of metals. In the laterite context, Clean TeQ has spent over 13 years developing and piloting the technology on laterite ores for optimal recovery of nickel, cobalt and scandium.<br>• To minimise technical risks, a well-resourced owner's management team with high levels of expertise gained from direct exposure to developed lateritic nickel projects, will be established. The resource and mining risk relating to the Project is deemed to be low because of the continuity of the deposit and the use of well-established resource estimation techniques. Resource and mining reserve estimates established for the first generation HPAL projects have proven to be reliable.<br>• Land access/ acquisition agreements will need to be finalised to cover parts of the Project area, limestone quarry, rail siding and natural gas pipeline. Additionally, the current water allocation will need to be increased or additional licences purchased to secure finalise the Project's water requirements. The water requirements are yet to be finalised and the final water access agreements are yet to be confirmed. |

| Risk/ Opportunity | Summary |
|---|---|
| Political | • The key permitting constraint identified by SRK for the Project relates to consents required to source and use water (whether surface water or groundwater) to satisfy operational water requirements. The primary Development Consent development consent required for project implementation was granted in May 2017, subject to a range of conditions, including completion of an array of pre-commencement studies and preparation of management plans. The pre-commencement conditions relating to the Project's water supply and water management will require a significant amount of technical work, stakeholder consultation and administrative effort.<br>• Political risks and opportunities relate to possible changes to the land tenure system, permitting requirements and royalty or taxation regimes. The risk of adverse changes being imposed by the Federal or New South Wales State Governments are considered low.<br>• The systems for granting land tenure and issuing permits for developing and operating mining and minerals processing plants are well established in New South Wales. With a granted Development Consent, development of the Project is subject only to financing and conversion of the Mining Lease Applications to Mining Leases. The Project site is in an area of no particular environmental significance and the Project has the support of Local and State Government leaders.<br>• Levels of royalty and taxation, and methods for their calculation, are also well established. The political climate in Australia at present is focused on accelerating development of the regional and rural areas where the economy is perceived to have suffered in recent years. |
| Commercial | • Commercial risks and opportunities relate to achieving the forecast product sales volumes and metals prices, as well as to the certainty of supply and price of major Project operating inputs.<br>• In order to mitigate the risk with respect to metal market movements, the Company commissioned an independent report on the nickel and cobalt markets by CRU Strategies (UK) and has consulted extensively with producers, consumers and traders in both metals.<br>• The strategy for minimising the risk of sales shortfall is to establish a relationship with a few off-take partner(s) who will off-take a large portion (80% - 100%) of the product. It should be noted that cobalt supply is largely from Africa and that the projected nickel laterite developments tend to fall off rapidly in cobalt production capacity within the first five years of operation. |
| Implementation | • Implementation issues include ensuring that the required technical standards in design and construction and that the Project is brought into production on or ahead of schedule. In general, the risk relates to failure of management to establish the optimum contracting strategy and structure and to monitor and control design and construction consultants and contractors to ensure that the Project is built to specifications, on time and within budget.<br>• An owner's team with a high level of expertise in process and engineering design, project controls and contracts establishment and administration, will be assembled. The team will have sufficient resources to establish and maintain a high level of monitoring and control. The personnel will be given responsibility levels sufficient to ensure optimal results. |
| Operations | • Operational issues which may constitute risks or opportunities include the recruitment and training of the required numbers of suitably qualified and experienced management, supervision and operations personnel, establishment of a suitable industrial relations structure for operations staff, awarding and administration of a maintenance contract, and ensuring that consumables and other supplies meet required specifications.<br>• The workforce establishment strategy pivots on the recruitment of an experienced operations management team and aggressive exploitation of the knowledge and experience gained from similar lateritic nickel operations currently operating. Early operator and maintenance personnel training programs are to be adapted for local employees and more experienced personnel from similar operations. |

# 26 Recommendations

Due to the positive economics SRK recommends that consideration be given to advancing the relevant aspects of the project to Feasibility Study level.

Specific recommendations are as follows:

Pit designs were not prepared as a basis for this Report. This report utilises the 2005 pit designs as it was considered that updated pit optimisation is likely to be similar to the 2005 Study results. SRK recommends that updated pit optimisations and pit designs be undertaken in the further study work to confirm the pit staging and optimise the Mineral Reserve, ore and waste schedules.

SRK recommends that updated pit optimisation be undertaken in the further study work to confirm the pit staging, limestone and waste schedules.

While the current project is reducing water demand, it is likely that supplementary water will be required. Alternative sources of supply may need to be considered if the appropriate approvals are not obtained in the future. SRK recommends that Clean TeQ finalise the water the water balance as part of the FS and secure any additional water requirement.

The pipeline operator and gas providers have advised that insufficient gas availability will not be a risk to the Syerston Project but pricing is less certain and is at risk of escalating prices. SRK recommends plans to secure gas supply contract are progressed.

SRK recommends that landholding agreements are finalised as soon as practical.

SRK recommends that all capital and operating cost estimates are reviewed and updated as required as part of the Feasibility Study

Prior to completion of the Feasibility Study, it is recommended that piloting the RIP system and downstream purification on fresh ore is carried out in order to produce samples and to confirm all process inputs for the full-scale plant

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Report Title: Syerston Nickel Cobalt Project, NI 43-101 Technical Report

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| Name/Title | Company |
|---|---|
| Ben Stockdale | Clean TeQ Holdings Limited |

| Rev No. | Date | Revised By | Revision Details |
|---|---|---|---|
| 0 | 01/11/2017 | Peter Fairfield | Final Report |