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

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Effective Date: 25 June 2018

Date and Signature Page

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Clean TeQ Holdings Limited
Sunrise Nickel Cobalt Project
New South Wales, Australia

NI 43-101 Technical Report

Project Manager: **Peter Fairfield**

Effective Date: **25 June 2018**

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Important Notice

SRK has prepared this Technical Report for Clean TeQ Holding Limited 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.

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.

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)

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Disclaimer

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
Ni	Nickel
NiEq	Nickel equivalent
AUD	Australian dollar
BEng	Bachelor of Engineering
BSc	Bachelor of Science
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Clean TeQ	Clean TeQ Holdings Ltd
Co	Cobalt
DTM	digital terrain model
FAusIMM	Fellow of The Australasian Institute of Mining and Metallurgy
GPS	global positioning system
GST	goods and services tax
g/t	grams per tonne
IRR	internal rate of return
kg	kilogram
kL	kilolitre
km	kilometre
kt	kilotonne
ktpa	kilotonnes per annum
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

Abbreviation	Meaning
mm	millimetres
mRL	metres reduced level
Mtpa	million tonnes per annum
m^3	cubic metres
MVA	megawatt ampere
mW	metres wide
Ni	Nickel
NiEq	Nickel equivalent
NI 43-101	National Instrument 43-101
NPV	net present value
oz	ounces
QA/QC	quality assurance/quality control
QP	Qualified Person
RC	reverse circulation
RO	reverse osmosis
ROM	run-of-mine
SD	standard deviation
SRK	SRK Consulting (Australasia) Pty Ltd
t	tonnes
tpa	tonnes per annum
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 owns the Clean TeQ Sunrise Project (the Project) in central New South Wales, Australia.

This Technical Report is based on review of, and work conducted as part of the Clean TeQ Sunrise Definitive Feasibility Study Report, (2018 DFS), the results of which were announced to the Australian Securities Exchange (ASX) and Toronto Stock Exchange (TSX) on 25 June 2018.

This Technical Report provides an analysis of mining and processing ore through a High-Pressure Acid Leach (HPAL) / continuous Resin-in-Pulp (cRIP) process, to determine the economics of producing hydrated nickel sulphate ($NiSO_4.6H_2O$) and hydrated cobalt sulphate ($CoSO_4.7H_2O$) and scandium oxide (Sc_2O_3).

The Mineral Resource Estimate (2017 Resource) was produced by independent consultants Widenbar & Associates, and Development & Mining Services (Kitto) in October 2017. This Mineral Resource was released to the Australian Securities Exchange (ASX) under the guidelines of the JORC Code (2012 Edition) and is re-reported in this Technical Report. The reported Mineral Reserve is based on the 2017 Mineral Resource Estimate

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

Table 1-1: Project – Key Parameters

Key Parameter	Basis of DFS	
Mineral Resource base	2017 Measured and Indicated Resources	
Mineral Reserve	2018 Probable and Proven Reserves	
Autoclave throughput	2.5 Mtpa	
Life of mine	40+ years	
Life of mine for financial modelling	25 years	
Average life of mine strip ratio	1.2:1	
	Years 2-11	Years 2-25
Autoclave Average Feed Grade		
- Nickel (%)	0.85	0.75
- Cobalt (%)	0.20	0.15
Average Metal Production		
- Contained nickel (tpa)	19,620	18,520
- Contained cobalt (tpa)	4,420	3,450
Average Sulphate Production		
- Nickel sulphate (tpa)	89,270	84,270
- Cobalt sulphate (tpa)	21,260	16,620
Average C1 Cash Costs/lb Nickel		
- Without by-product credits (USD/lb)	4.71	4.68
- With by-product credits (USD/lb)	-2.57	-1.46
Average Recoveries		
- Nickel (%)	92.6	

Key Parameter	Basis of DFS
- Cobalt (%)	91.2
Long Term Price Assumptions	
- London Metals Exchange nickel	USD7.00/lb nickel
- Nickel sulphate premium	USD1.00/lb nickel (contained)
- London Metals Exchange cobalt	USD30.00/lb cobalt
- Cobalt sulphate premium	Nil
Total Development Capital Cost	USD1,491M (AUD1,988M)
USD:AUD exchange rate	0.75
Net present value (NPV_8) (real, post tax)	USD1,392M
Internal rate of return (real, post tax)	19.1%
Simple payback (real cash flow)	4.3 years

The Project will produce up to 25,000 tpa of nickel and up to 7,000 tpa of cobalt, as high-purity sulphates for the rapidly expanding lithium-ion battery market. The project will also produce 10 tpa of scandium oxide (with the capacity to produce up to 80 tpa if sufficient demand exists), to become one of the world's only large scale sustainable sources of supply. The scandium sales volume, capped at 10 tpa, represents 3% of the total project revenue. The assumed price for scandium is USD1,500/kg.

The Project net present value (NPV_8) (real, post tax) is projected at USD1,392M for an initial capital cost expenditure of USD1,491M, delivering a (real, post-tax) internal rate of return (IRR) of 19.1% and a simple payback period of 4.3 years. First product shipment is targeted for quarter one, 2021 with a total projected mine life exceeding 40 years.

Given the substantial cobalt credits generated by the Project, the average life of mine operating costs will be USD-1.46/lb nickel (including by-product credits).

The Project will become a low cost, large scale producer of high-value metals for rapidly growing technology markets. It will produce three products from the one mine.

The project benefits from several key competitive advantages:

- **Three high-grade products from one mine** – provides protection from single commodity price cycles and provides flexibility to vary product mix to suit market demand
- **By-product credits** – exceptionally high by-product credits, especially from cobalt
- **Low-cost, proprietary process** – combining traditional hydrometallurgical flow sheet with Clean TeQ's proprietary ion exchange technology optimizes metal recovery, for lower capital and unit operating costs, much higher purity product and a smaller process plant footprint
- **Access to infrastructure** – close to road and rail infrastructure connecting the Project to major sea freight corridors to international suppliers and to Asian markets
- **Low sovereign risk** – Australia is seen as one of the world's most stable and low risk countries for resources development, offering transparency and credibility to supply chain audits.

1.1.1 Study methodology

The Project studies have progressed significantly – 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 Sunrise (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 Mineral Resource Estimate (2017 Resource) was produced by independent consultants Widenbar & Associates, and Development & Mining Services (Kitto) in October 2017.

1.2 Project overview

1.2.1 Project location

The Project is located near the town of Fifield in central New South Wales, approximately 350 km west of Sydney, Australia. The Project site is well serviced by existing infrastructure inclusive of sealed roads and lies in close proximity to rail, power and regional population centres. The local town is Fifield which is located 4 km from the Project area, (Figure 1-1).

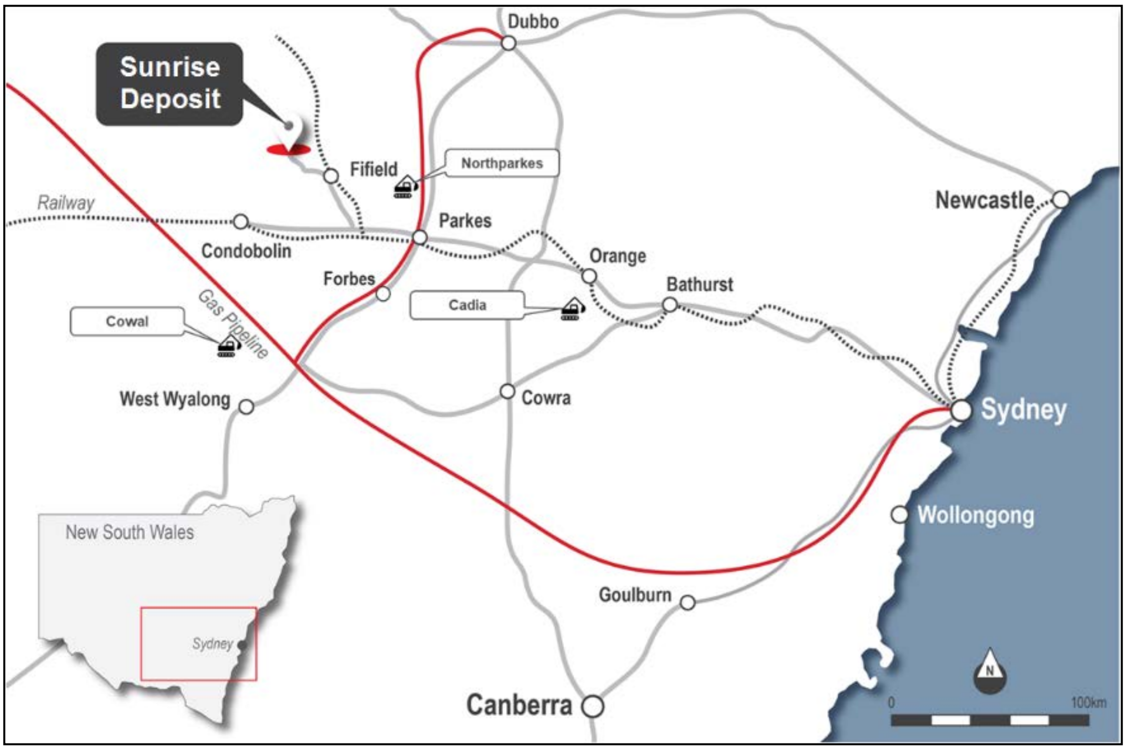

Figure 1-1: Project Location

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.

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).

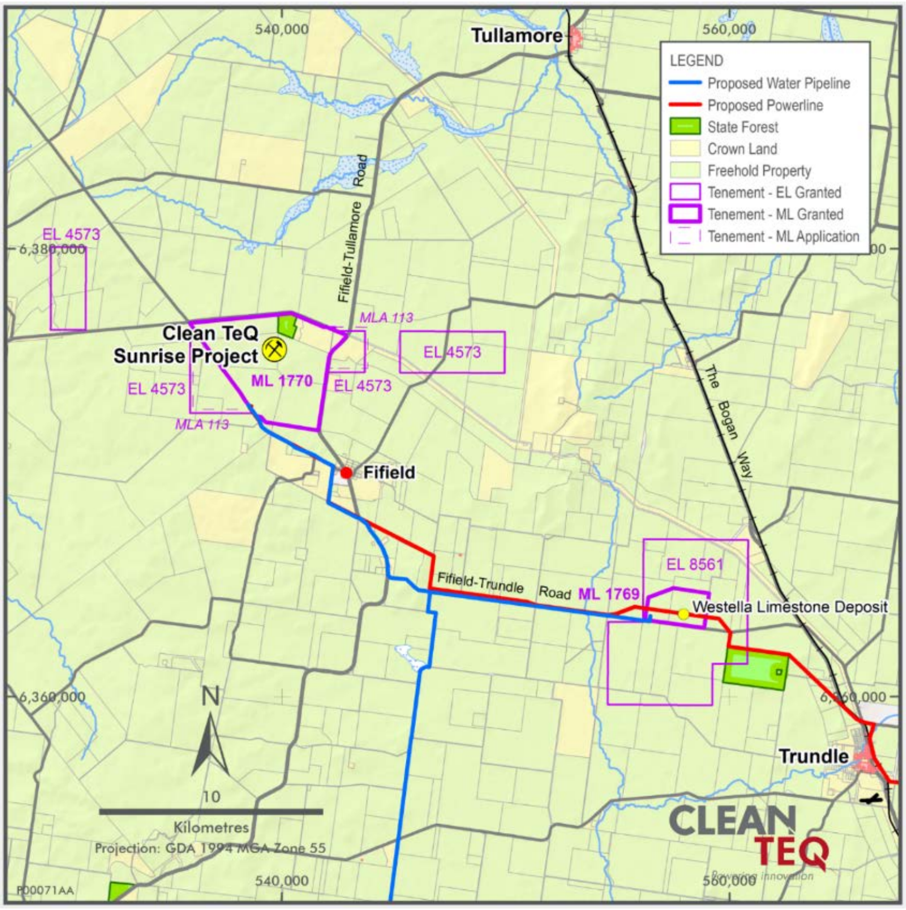

Figure 1-2: Tenement Overview

One of the Project's competitive advantages is its proximity to existing infrastructure. The Project is located ~30 km from a source of high voltage power from the electrical grid, a rail line within 25 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 project is 100% owned by Clean TeQ Holdings Limited through its wholly owned subsidiary, Clean TeQ Sunrise Pty Ltd.

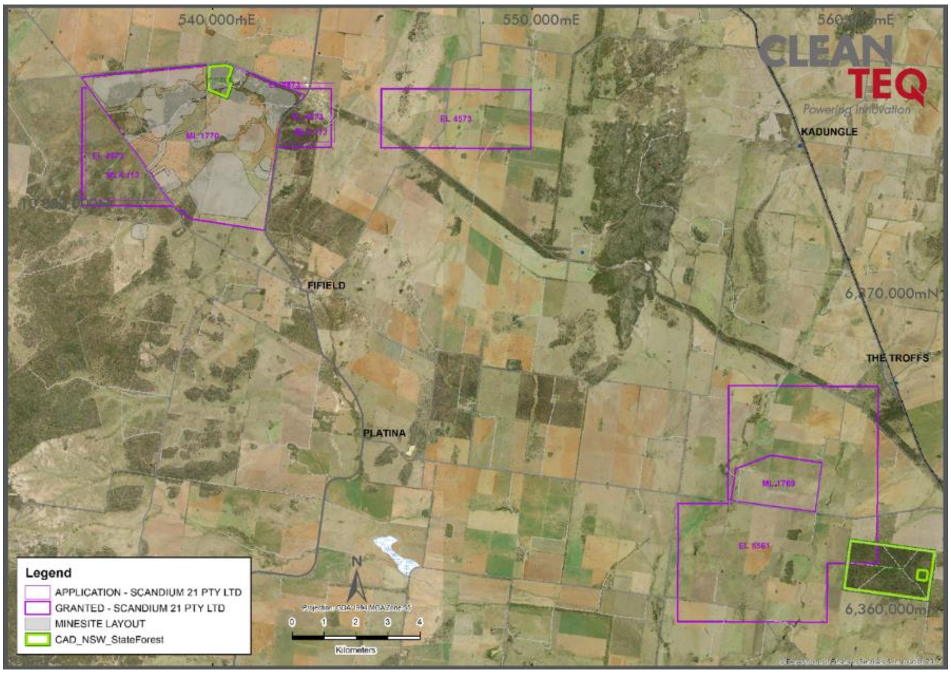

Figure 1-3: Aerial Photograph of Tenement Ownership Overview

1.2.2.1 Mineral Tenure

The main project area is covered by two adjoining tenements, Mining Lease ML 1770 and a portion of Mining Lease Application MLA 113. These two tenements are underlain by Exploration License EL 4573. Mining Lease ML 1769 covers the Westella limestone deposit which is underlain by Exploration License EL 8561. All of the Mining Leases, Mining Lease Applications and Exploration Licenses covering the main project area are 100% controlled by Clean TeQ Sunrise Pty Ltd.

1.2.2.2 Exploration Licenses

Exploration Licenses EL 4573 and EL 8561 have been granted subject to the provisions of the New South Wales Mining Act 1992 and to the conditions of each License.

Table 1-2: Exploration License Details

License Number	Date of Grant	Expiry Date	Area Granted	Current Area (ha)	Ownership (%)	Annual Expenditure AUD)
EL 4573	17 Aug 1993	16 Aug 2018 Renewed 3rd August 2018	18 units	3856	100	50,000
EL 8561	11 May 2017	11 May 2020	13 units	3744	100	25,000

1.2.2.3 Mining Leases and Mining Lease Applications

The boundaries of the granted Mining Leases ML 1769 and ML 1770 and pending Mining Lease Application MLA 113 which have been applied for over the Project area and Westella limestone deposit are shown in Figure 1-4 and Figure 1-5. The status of each Mining Lease and Mining Lease Application is indicated in Table 1-3.

Table 1-3: Project Mining Lease Applications

License Number	Area	Application/ Grant Date	Grant Status	Expiry Date	Interest (%)
ML 1769	389.7ha	15 Feb 2018	Granted	15 Feb 2039	100
ML 1770	2,195.0ha	16 Feb 2018	Granted	16 Feb 2039	100
MLA 113	8 units	10 August 1998	Pending	Pending	100

1.2.2.3 Land Holding and Surface Rights

Except for the parcels of land covered by the State Forest and the Crown Land, all of the land covered by the main project area is 100% controlled by Clean TeQ Sunrise Pty Ltd.

Figure 1-4 and Figure 1-5 show the landholding the Project area and limestone deposit. Under the current Development Consent, surface rights are only restricted in the State Forest parcel of land. The Crown Land parcel can be accessed for the Project needs.

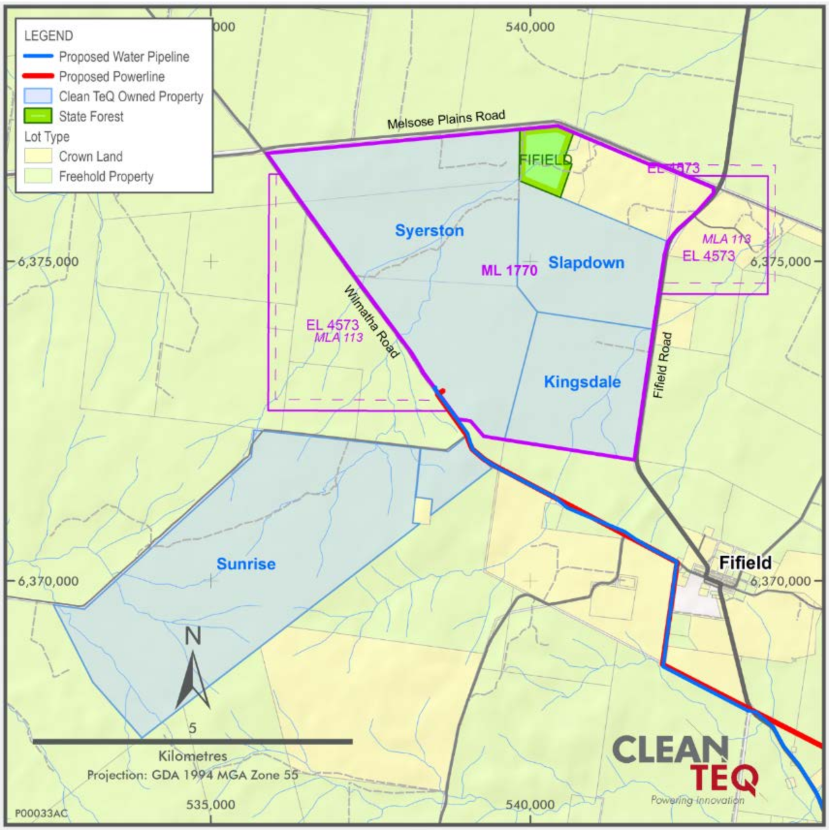

Figure 1-4: Clean TeQ Sunrise Pty Ltd Landholding – Main Project Area

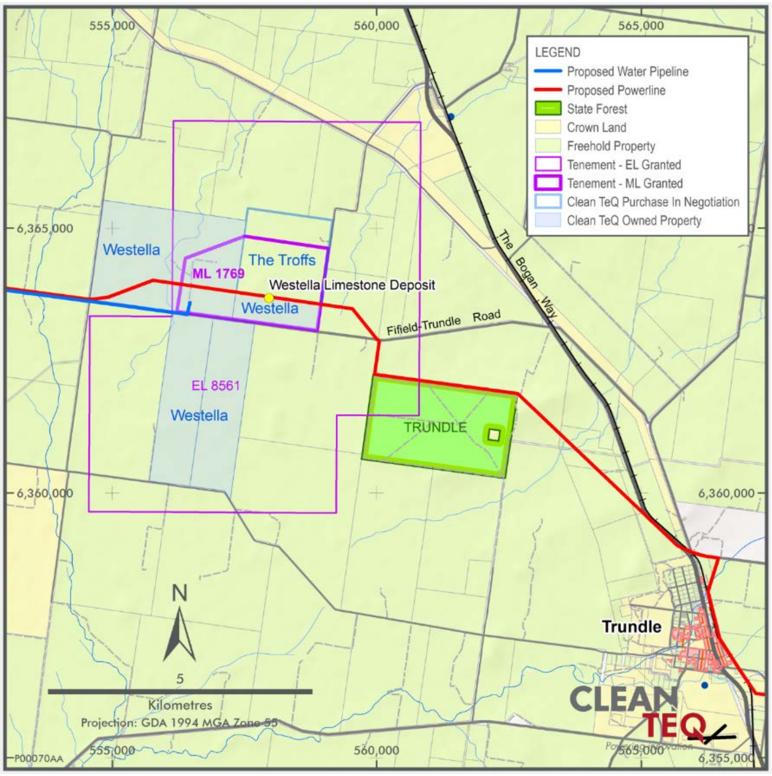

Figure 1-5: Clean TeQ Sunrise Pty Ltd Landholding – Westella Limestone Quarry

1.3 Geology and Mineral Resource

1.3.1 Summary

The deposit is a large, flat, near-surface, continuous, nickel laterite deposit, with associated cobalt and local elevated platinum and scandium. A mineralised layer up to approximately 40 m thick covers an area about 4 km x 2 km and lies 0 – 30 m beneath the surface. It thins at the edges presenting a basin like form, Figure 1-6.

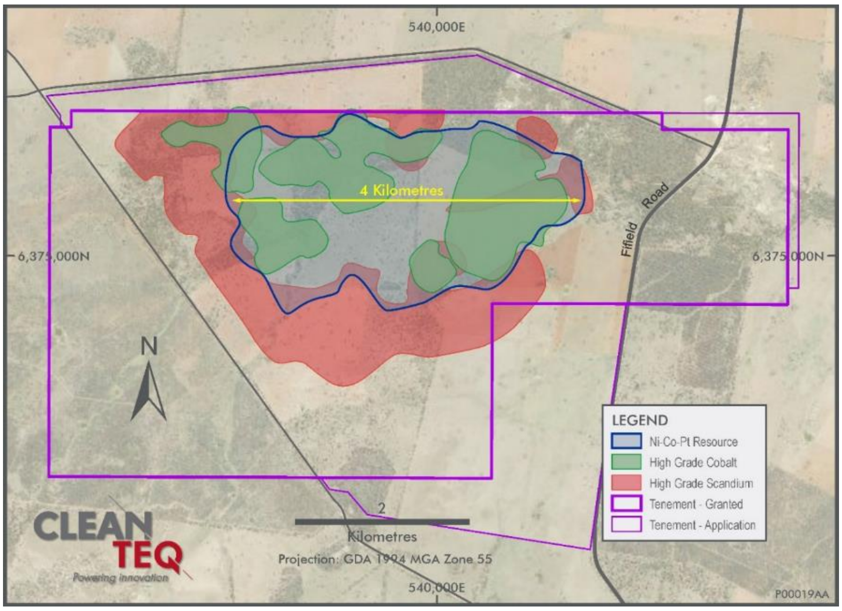

Figure 1-6: Plan Showing Distribution of High-Grade Resources

1.3.2 Geological overview

The deposit is an iron rich oxide type nickel laterite deposit, with higher than normal levels of associated cobalt and locally elevated platinum and scandium.

The deposit has developed over an ultramafic intrusive complex. The laterite sits over a serpentinised dunite core (dunite complex) and thins out over peripheral pyroxenites. The laterite profile is partly overlain by transported alluvium, Figure 1-7.

Interpretation of the laterite profile has indicated five geochemically-distinctive, sub-horizontal zones (LATZONES):

- Residual Overburden
- Transitional Zone
- Goethite Zone
- Silicified Goethite Zone
- Saprolite Zone.

Nickel and cobalt mineralisation are best developed in the Goethite Zone and extend into the Silicified Goethite Zone. Highest nickel and cobalt values occur over the dunite complex.

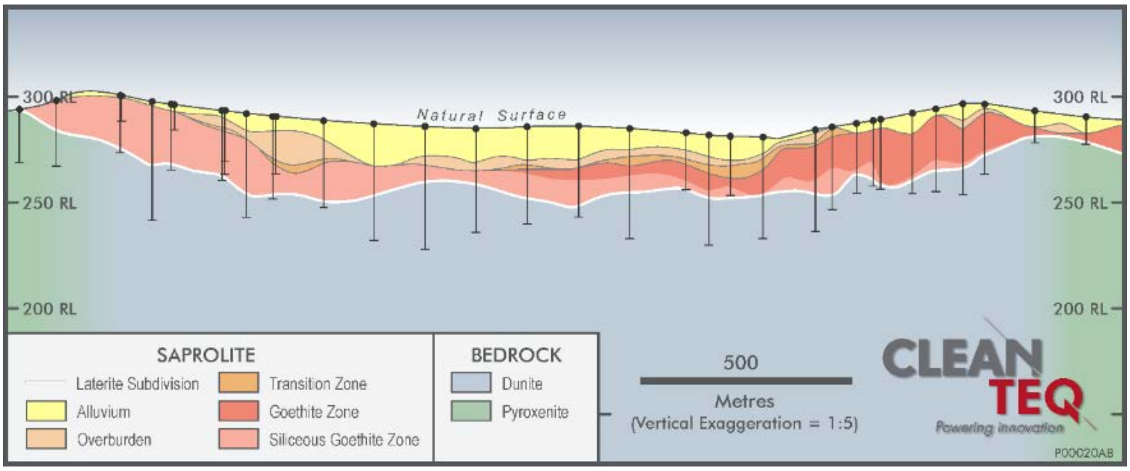

Figure 1-7: Schematic Geological Section

The alluvial cover varies from 0 m to >10 m thick. The overburden zone is commonly characterised by clay, gravel and ironstone. The Transition Zone varies up to about 15 m in thickness.

The Goethite Zone is a relatively uniform, very fine grained, orange-brown, goethite rich layer up to about 20 m thick. It typically contains more than 40% iron and 5-10% silicon. The silicified Goethite Zone is typically 10-30 m thick but can be extremely variable. It includes approximately 20% silicon.

1.3.3 Resource Model - Mineral Resource Modelling and Estimation

1.3.3.1 Drilling, Assaying and Metallurgical Sampling

The 2017 Mineral Resource Estimate is based on 1,318 reverse circulation drill holes (48,459 m) and 45 air core holes (1,441 m), which were drilled between August 1997 and November 2015. Material for metallurgical testwork and bulk density measurements was provided by a total of 13 shallow, vertical diamond core holes (drilled between 1997 and 2000) and nine large diameter (770 mm) Calweld holes which were drilled in 1999.

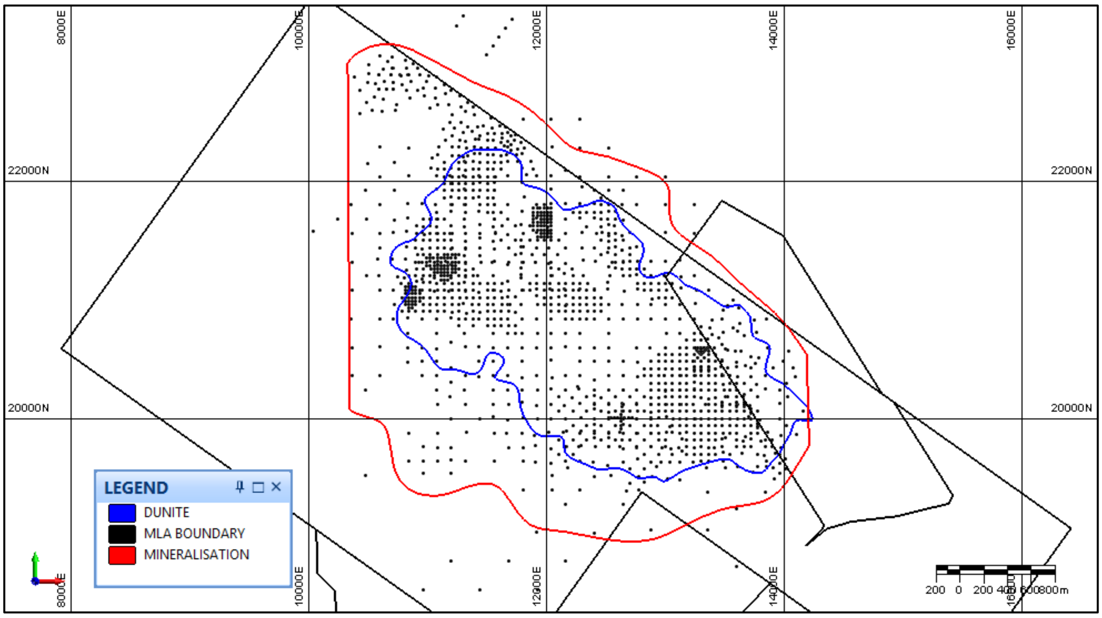

Figure 1-8: Drill Hole Location Plan

Data preparation included validation of geological logging, drill hole collar surveys and coordinate transformations, retrieval of historical assays and assay quality assurance/quality control (for standards, duplicates and replicates). A separate program of independent custody sampling (using samples from twin reverse circulation and geology from diamond core holes) was carried out between September and November 2017, to test ten reverse circulation drill holes from historical resource definition drilling.

Input data was a validated Micromine™ database, with validated collar and assay data sourced from original survey and laboratory files where possible. The drill hole data files are currently stored in a high-level relational database management system (Microsoft SQL Server 2016). Drill hole data is stored in a drill hole database model (CLQ_MODEL) which has been configured within the Micromine™ Geobank™ database management program. This initiated the creation of the various data tables which constitute the Geobank™ data model.

Historical assays were imported into Geobank™ from original source datasets and quality assurance/quality control validation carried out against assays for original standards and duplicates.

A full revision of the geological interpretation of the LATZONES was carried out in 2017, using 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. This approach indicated that the LATZONES have distinctly different chemical signatures, which can be used in conjunction with geological logging data to subdivide the laterite profile.

1.3.3.2 Geological Modelling

The geological modelling process used the geological interpretation as the framework for creating new geological surfaces. Geological interpretation strings on north-south cross sections were linked by tie-lines and various triangulation processes were used to create digital terrain models.

Once the digital terrain model surfaces had been validated they were made available for Resource Modelling, where the entire regions between the zones were filled with blocks to create a rock model.

Statistical analysis for the main elements of interest was carried out on 1 m composited data, which was coded by geological zone (LATZONE). Assay distribution analysis was also completed by domain. Interfaces for major domains were reviewed for the major elements to be estimated and it was concluded that hard wiring of LATZONE boundaries was appropriate. Correlation analysis and variography were also completed.

Resource Estimation was carried out using Micromine™. Initially a three-dimensional rock model was generated with subdivision based on LATZONE domains, the outline of the dunite complex and an outer mineralisation limit. Block grades estimated for cobalt and nickel, as well as scandium, platinum, iron, silicon, magnesium, aluminium and manganese, were generated using Ordinary Kriging.

Appropriate block sizes in the model were selected according to drill spacing. Wireframe surfaces were smoothed by the implicit surface modelling process within Micromine™, which generates smooth surfaces while honouring snapped drill hole points. Wireframes were filled above or below with blocks and combined in a controlled sequence to generate a three-dimensional rock model.

Search parameters were based on the variogram ranges. Search ellipses are generally isotropic, and a flattening/ unfolding process was used to simplify the specification of the slight variations of dip needed to follow the undulations in the surfaces. A high-grade cobalt domain (above 0.15% cobalt) was defined within the Goethite Zone using an indicator modelling approach. A similar indicator modelling approach was used to model high-grade scandium, but this was restricted to areas outside the dunite complex footprint.

Bulk density factors used for previous Mineral Resource Estimates have been used in this update and have been assigned by geological domain. Dry density values have been derived for all geological domains by the following methods:

- volumetric measurements on small core samples from five diamond core holes, with weights determined after drying samples
- water immersion measurements on one diamond core hole
- bulk density calculations on total material obtained from over 100m of large diameter Calweld drilling. These samples were quickly sealed on collection and adjusted for moisture after oven drying
- down hole gamma-gamma geophysical logging of seven diamond core holes and around 137 reverse circulation drill holes by Down Hole Surveys Pty Ltd or Surtron Technologies Pty Ltd.

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 Mineral Reserves (JORC Code). A summary of the 2018 DFS Resource Estimates follows.

The Project’s large nickel laterite deposit is associated with unusually high-grade cobalt and is surrounded by one of the largest high-grade scandium deposits in the world.

Work is currently underway to assess options for the recovery of platinum (in concentrate form) from the front end of the plant flow sheet. The extraction and sale of a platinum concentrate represents a future additional revenue opportunity for Clean TeQ.

1.3.3.3 Mineral Resource Estimate

This Mineral Resource Estimate that forms the basis for mine planning and Mineral Reserve Calculations, as detailed in Chapter 15 and 16, is presented in Table 1-4 and Table 1-5. The scandium estimate is presented in Table 1-6.

Table 1-4: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate at 0.06% Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	40	0.15	0.75	59,000	298,942
Indicated	47	0.12	0.55	58,000	259,479
Measured + Indicated	**87**	**0.13**	**0.64**	**116,000**	**558,421**
Inferred	14	0.11	0.24	16,000	34,643

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 1-5: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate - No Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	69	0.10	0.63	69,000	436,000
Indicated	94	0.08	0.47	75,000	438,000
Measured + Indicated	**163**	**0.09**	**0.54**	**144,000**	**874,000**
Inferred	21	0.09	0.23	18,000	48,000

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 1-6: Clean TeQ Sunrise Scandium MRE – (No Scandium Cut-Off) TZ, GZ and SGZ Only

Scandium Cut-Off	Classification	Quantity (Mt)	Scandium (ppm)	Scandium Metal (t)	Sc_2O_3 (t)
No Cut-Off TZ, GZ and SGZ only.	Measured	68.81	62	4,238	6,500
	Indicated	93.90	86	8,096	12,417
	Meas + Ind	162.70	76	12,334	18,918
	Inferred	20.62	283	5,829	8,940

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
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1.4 Mining and Mineral Reserve

1.4.1 Mineral Reserve

The deposit contains nickel, cobalt, platinum and scandium and is planned to be mined using open pit mining methods.

This Mineral Reserve include the economically mineable part of the Measured and Indicated Resource. It includes mining dilution and allows for mining losses. Appropriate assessments and studies incorporate realistically estimated mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate that, at the time of reporting, extraction can reasonably be justified.

Total current Mineral Reserves, presented in Table 1-7 support a mine life exceeding 40 years. The pit design includes 0.3 Mt of inferred resource, reported as waste and not included as part of this Mineral Reserve Estimate.

Scott McEwing 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, Scott McEwing takes responsibility for the reporting of the Mineral Reserves in this Report.

1.4.2 Pit Model and Mine Schedule

The open pit mine design work included:

- mine scheduling
- mine design.

Several mine optimisation runs were undertaken. Optimisation considered higher-grade cobalt and nickel feed during initial mining to maximise revenue and economic value.

Strategic scheduling was completed using NPVS and Multimine, which use mixed integer-linear programming to find the pit/phase/bench extraction sequence that maximises an objective function, usually project discounted cash flow. The NPVS/ Multimine package manages most mining and metallurgical parameters and constraints simultaneously in the optimisation, (e.g. plant capacity, mining capacity, grade/metallurgical constraints, and long-term low-grade stockpile and reclaim strategies).

The initial NPVS pit optimisation shells were used as a guide for pit design. The updated pit designs were then used as a basis for the final strategic scheduling runs in NPVS/ Multimine

A number of mine schedules and scenarios were developed. They include:

- higher-grade, and increased mining movement, with throughput into the PAL at 2.5 Mt ore per annum after ramp-up (mining constraint into eastern side of the deposit)
- smoothed metal production with throughput into the PAL at 2.5 Mt ore per annum, with constraint of pit to the current Mod 4 Development Consent Approved pit boundaries followed by expansion into the ultimate pit design.

Ultimate pit designs were developed based on NPVS optimal pit shells assuming conventional selective mining methods utilising excavators, haul trucks coupled to a ROM dumping point.

Table 1-7: Clean TeQ Sunrise Cobalt/ Nickel Open Pit Mineral Reserve Estimate

Classification	**Inventory (Mt)**	**Grade (% Ni)**	**Grade (% Co)**	Grade (ppm Sc)
Proven	65.5	0.645	0.103	48
Probable	81.9	0.493	0.084	57
Proven + Probable	147.4	0.561	0.092	53

Notes:

1) Tonnes are rounded to the nearest thousand.
2) Totals may appear different from the sum of their components due to rounding.
3) The Proved Mineral Reserve estimate is based on Mineral Resources classified as Measured, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine. The Probable Mineral Reserve estimate is based on Mineral Resources classified as Indicated, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine.
4) The Mineral Reserve is inclusive of Mineral Resources
5) Only Measured and Indicated resource classifications are considered in the estimate.
6) The Mineral Reserve estimate was independently verified by Scott McEwing, FAusIMM, CP (Mining), who is a full-time employee of SRK Consulting and a Qualified Person under NI 43-101.

The high by-product credits (Scandium and Ammonium Sulphate) provide an opportunity for a high unit-revenue to mining-cost ratio that drives mine optimisation towards high-grading and high material movements.

The approach to mining the deposit has been strongly influenced by the need to avoid problems experienced by previous nickel/ cobalt projects, notably inconsistent ore grade, inconsistent ore quality, and difficult ore handling.

The consistency of ore feed quality to the pressure acid leach plant is paramount. The relatively low mining cost compared to the processing cost drives a focus on plant unit costs and productivities rather than on mining efficiencies and mining costs. A tight grade-control process, a selective mining method, and a schedule based on multiple open faces, direct tipping, and active ore blending strategies are more important than mining efficiencies.

With these objectives in mind, significant work has been carried out on the deposit with two key aims:

- understand and quantify the scale and distance over which nickel, cobalt and the acid consuming elements are distributed in the orebody
- devise a grade control and blending system that allows mining to reliably deliver ore feed within the variability control limits required for processing.

The resource work identified significant grade variability changes at the sub-10 m scale, especially in the cobalt, and has significantly increased the ability to target high metal grades early in the mine life. With such a degree of variability, when drill spacing is too large the actual grades may be significantly more variable than the predicted grades. Closer and appropriately spaced grade control drilling will enable ore blocks of varying qualities to be more accurately estimated, mined simultaneously and blended in the ore preparation circuit.

The options of owner-operated mining and contract mining were considered. The use of contractors would primarily diversify risk and minimise start-up capital costs. This option would enable Clean TeQ to start the project with an experienced and trained contractor workforce and contractor systems, thus minimizing the distractions of:

- recruiting and training mine operations personnel
- establishing effective operating and management systems
- procuring the earthmoving fleet and establishing the mine facilities.

The base-case assumes the use of contract mining but also recognises the requirement for Clean TeQ to maintain an active role in day to day management of the mining activities.

The mining capital cost estimate, Table 1-8, is based on the use of a mining contractor and includes estimates for pre-mining works and construction of mining facilities.

Table 1-8: Total Capital Costs by Area/Activity

Mining Activity	Qty	Unit	Rate (AUD '000)	Total (AUD '000)
Mobilise/demobilise	1	Sum	900	900
Establishment	1	Sum	2,960	2,960
Clearing	1,233	ha	1.74	2,149
Topsoil removal	1,233	ha	11.34	13,991
Turkey's-nest	1	Item	160	160
Main truck - haul road sheeting	13	km	91	1,227
Grade control (year 0)	59,286	Lm	25.98	1,540
Total				**22,928**

1.4.3 Mine Operating Costs Over Life of Mine

A detailed mining cost model and mining cost estimate was prepared by independent mining consultant (IQE Pty Ltd). Key aspects of this cost estimate are shown in Table 1-9.

Table 1-9: Summary of Life of Mine Mining Costs[1]

Mining Activity	Qty (000)	Unit	Rate (AUD)	Total (AUD '000)
Load and haul - waste to dumps	164,620	DT	2.47	407,121
Load and haul - ore to run-of-mine	17,013	DT	7.07	120,284
Load and haul - ore to stockpile	126,303	DT	5.12	646,216
Load and haul - reclaim from stockpile	52,089	DT	2.47	128,580
Subtotal Load and Haul	**360,026**	**DT**		**1,302,201**
Establishment	1	Sum	-	2,960
Mobilise/demobilise	1	Sum	-	900
Subtotal Establishment Items				**3,861**
Crusher feed	34,551,154	DT	0.85	29,397
Drill and blast SGZ material	10,540	BCM	1.90	20,026
Grade control drilling	1,541	LM	25.98	40,040
Subtotal Additional Mining Items	-	-	-	**89,463**
Civil Items	-	-	-	17,527
Total	-	-	-	**1,413,052**
Total (AUD/t Material Movement)	**3.92**			
Total (AUD/t Ore Feed)	**20.45**			

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[1] No allowances for inflation, all money in current dollars as of June 2018.

Key parameters used as part of the pit optimisation work to determine the pit extents included (but are not limited to):

- an average of 2.5 Mtpa ore into the autoclave
- a nominal 25 year mine life with ramp-up constraints
- throughput limits of 25,000 tpa of recoverable nickel and 7,000 tpa of recoverable cobalt, with combined metal maximum of 30,000 tpa
- variable ROM ore feed into the processing plant varying from 2.5-3.4 Mtpa of ore based on beneficiation circuit upgrading of ore
- revenue was determined from recovered nickel metal and nickel price assumptions (USD7.5/lb) and recovered cobalt metal and cobalt price assumptions (USD14/lb). Higher cobalt price assumptions were used in the financial model
- mining costs used from the pre-feasibility study produced in 2016 were used for initial mining optimization/ scheduling, then updated mining costs produced by independent mining cost estimators, IQE as part of the DFS Mining work and financial modelling.

Figure 1-9 shows the nickel and cobalt production volumes resulting from these parameters.

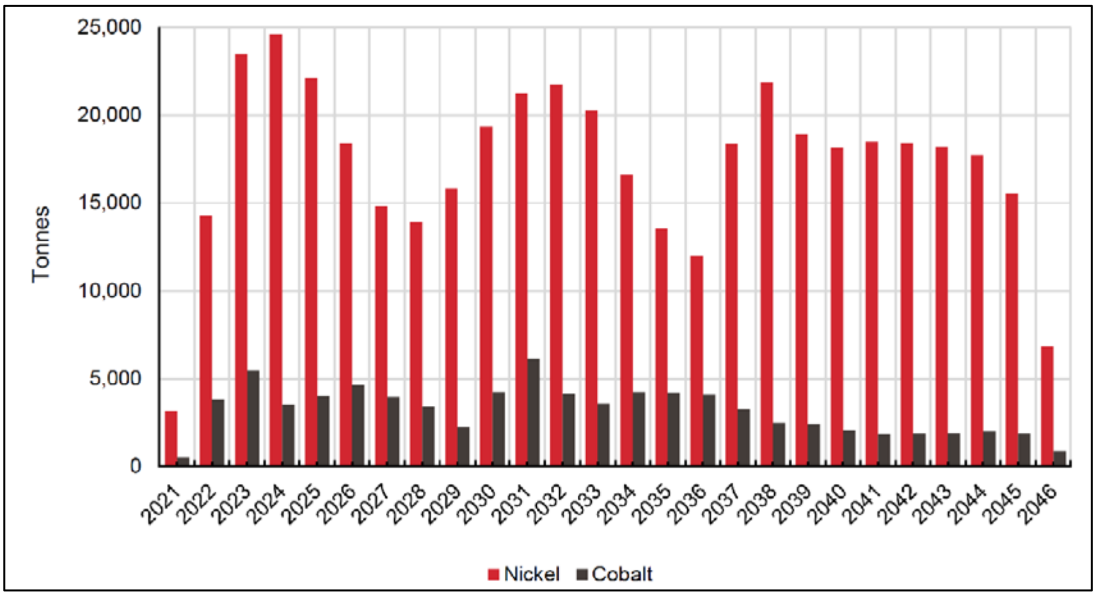

Figure 1-9 Mine Plan (Lower Total Material Movements Case)

As a post-DFS opportunity, the high intermediate-grade stockpile and reclaim tonnages and the associated mining costs, could be reduced significantly by lowering the cut-off grade of ore feed to the PAL to utilise the full potential of the larger-than-design autoclaves. Some design provisions would have to be allowed for at the front-end of the process plant, from ore preparation to partial neutralisation.

1.4.4 Limestone supply

Limestone feed is required for acid neutralisation at the plant. The acid neutralisation design and operating costs for the Project are based on a delivered limestone grade of 90% $CaCO_3$.

There are a number of potentially viable sources of supply of Limestone in the region are presented in Table 1-10:

Table 1-10: Sources of Limestone

Supplier	Grade ($CaCO_3$ %)	Estimated Tonnage p.a.	Distance km
Westella	43%	920,000	20
Westlime	95%	416,421	70
Ezylime	95%	416,421	70
Cudal	95%	416,421	140
Molong	95%	416,421	150

The Westella limestone deposit is located approximately 20 km southeast of the proposed 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 2018 DFS.

At Westella, high grade limestone would be crushed and stockpiled at the quarry ready for transport by a mining contractor. Low-grade limestone and waste material would 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. Clean TeQ has indicated that the resource continues to the north, but further drilling is required to prove this.

The Widenbar evaluation notes that at a cut-off grade of ~80%, a limestone supply life from Ezylime at a limestone solids grade of 90% $CaCO_3$ is > 15 years, based on, and limited by, resource drilling to date.

This resource has not yet been classified in accordance with the 2012 Australasian code for the reporting of Mineral Resources and Mineral Reserves (JORC Code).

1.5 Mineral Processing Overview

A hydrometallurgical processing flowsheet using conventional HPAL to leach nickel and cobalt from the Project ores will be used at the Processing Plant. The leached nickel and cobalt is recovered through nickel and cobalt cRIP from where the nickel and cobalt eluate is processed via solvent extraction (SX) before the final nickel sulphate and cobalt sulphate products are crystallised, dried, packaged and transported to market.

Scandium oxide will be produced as a by-product, using continuous liquid ion exchange (cLX) to extract scandium from the nickel and cobalt eluate before scandium refining and final calcination.

The process plant will produce high purity hydrated nickel and cobalt sulphate products, as well as a 99.9% Sc_2O_3 product. Figure 1-10 shows an overview of the processing options considered for the 2018 DFS.

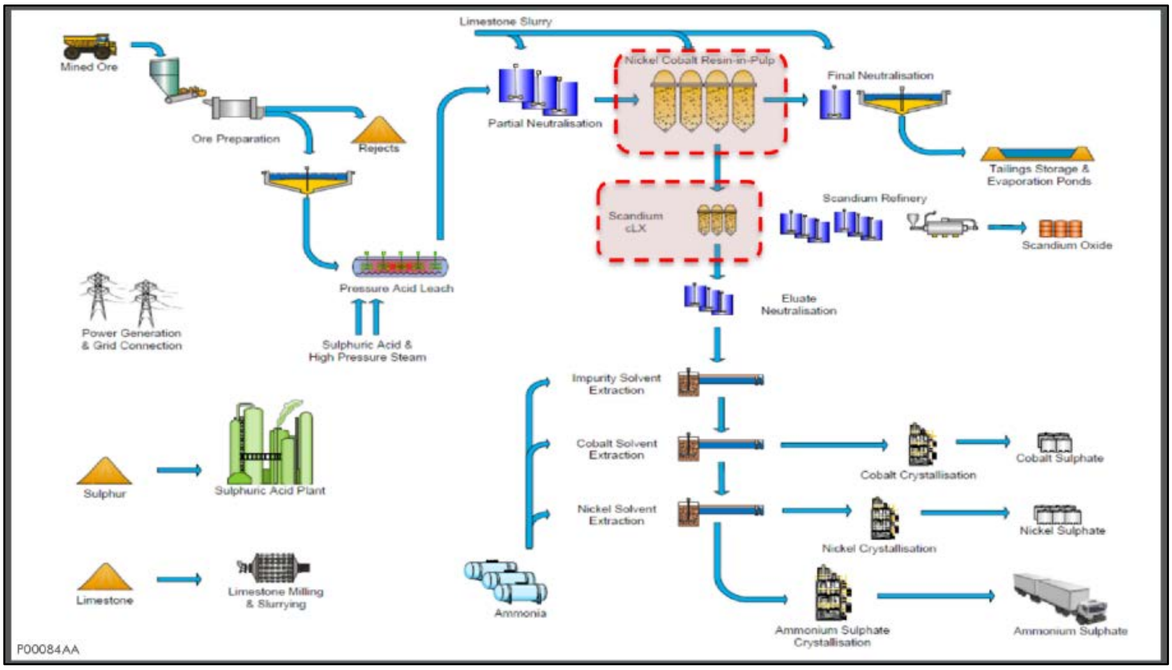

Figure 1-10: Process flowsheet

The process can be broadly defined as:

- Ore Preparation and Milling, comprising a sizer and a mill. Mill discharge is classified by screening, with the coarser lower grade quartz particles being rejected before thickening.
- HPAL. Slurry from the storage tanks is heated before flowing into the pressure acid leach autoclaves and mixed with concentrated sulphuric acid under high temperature and pressure. The leached slurry is then discharged from the autoclave through three flash vessels and is partially neutralised.
- Nickel Cobalt cRIP. Partially neutralised slurry passes through the nickel/cobalt continuous resin-in-pulp adsorption circuit where the nickel, cobalt and scandium and some impurity metals are collected by the resin. The resin undergoes desorption with dilute sulphuric acid where the nickel, cobalt and scandium are eluted from the resin into a high-grade, low-impurity-rich liquor. Depleted slurry is discharged to the tailings stream where it is neutralised, thickened and pumped to the tails.
- Scandium cLX and Refinery. The scandium liquid ion-exchange collects scandium from the nickel/cobalt eluate and pumps it to the scandium refinery for processing into a saleable high-purity scandium oxide product.
- Nickel/ Cobalt Sulphate purification and recovery. The nickel/cobalt eluate post-scandium cLX is further neutralised and processed via solvent extraction for impurity removal, cobalt separation and nickel separation before ion exchange polishing and crystallisation to produce high purity nickel and cobalt sulphates
- Tailings neutralisation and storage. Tailings will be pumped to the Tailings Storage Facility (with total footprint area of 380 ha) as a slurry with a solids concentration of approximately 42% w/w.
- Process reagents and utilities (sulphuric acid, steam, water, limestone, other).

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 commissioning and ramping up HPAL operations is challenging, 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 Ramu 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 the Project ore demonstrates high nickel and cobalt extractions (approx. 94% - 96% each respectively), while scandium extraction using the HPAL process ranges from 80% - 90% and is sometimes higher.

1.5.2 Continuous Resin-in-Pulp

Clean TeQ has incorporated a proprietary ion exchange technology (Clean-iX®) for the extraction and purification of metals to replace counter-current decantation and subsequent downstream refining requirements to produce high-purity products by a more direct process route.

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 ten additional patents on various aspects of the technology, including one for extraction and purification of scandium.

The application of continuous Resin-in-Pulp (cRIP) for nickel, cobalt and scandium recovery is based on commercially applied (for other metals, notably gold and uranium) developed equipment and technologies, and has the potential for significant benefits for nickel, cobalt and scandium recovery.

Conventional industry flowsheets currently use counter-current decantation (CCD), followed by precipitation of either a mixed sulphide or hydroxide intermediate, before re-leaching and solvent extraction (SX) to both remove impurities and separate nickel and cobalt before final product. This process has several disadvantages, including higher capital and operating costs, and lower metal recoveries.

The use of cRIP technology addresses many of these issues. The cRIP method uses solid ion-exchange resin beads which are contacted directly with the leached slurry, resulting in extraction of more than 99% 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 clear 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 cRIP 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 had 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 recent metallurgical testwork and processing development focus has been to:

- apply cRIP technology to the Project
- change the previous Project Owner's flowsheet such that the HPAL discharge slurry incorporates refining nickel and cobalt to high-purity hydrated nickel and cobalt sulphates.
- demonstrate the ability to produce high-purity scandium oxide

The Project ore has 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 – 95.7% for nickel and 95% for cobalt – and higher overall metal recoveries of 92.6% and 91.2% respectively.

1.6 Processing plant

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 desorption circuit of the cRIP plant is a high-concentration nickel and cobalt sulphate solution with low impurities. 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.

The Project will produce the following products:

- Nickel sulphate – primarily for the lithium-ion battery market
- Cobalt sulphate – primarily for the lithium-ion battery market
- Scandium oxide – A developing market but high-value as an alloying material for lightweight aluminium alloys.
- Ammonium sulphate This will be sold primarily as an agricultural fertiliser.

The annual production capacity of each product will vary based on the mine planning optimisation:

Table 1-11: Average Production

Description		Units	Quantity
Life of Mine / Reserves duration		years	>25 / 40
Nickel Sulphate ($NiSO4.6H2O$)	(Years 2 – 11)	tpa	89,270
Cobalt Sulphate ($CoSO4.7H2O$)	(Years 2 – 11)	tpa	21,260
Nickel Equivalent Production	(Years 2 - 6) (Years 2 – 11) (Years 2 - 25)	tpa	21,780 19,620 18,520
Cobalt Equivalent Production	(Years 2 - 6) (Years 2 – 11) (Years 2 - 25)	tpa	4,640 4,420 3,450
Scandium Oxide (Sc_2O_3)		tpa	~ 80
Nickel Grade (Average Years 2 - 6)		%	0.92
Cobalt Grade (Average Years 2 - 6)		%	0.20
Ammonium Sulphate		tpa	82,000

Table 1-12: Contribution to Total Project Life of Mine Revenue by Product Type

Product	Revenue (USD)	Proportion of Revenue (%)
Nickel sulphate	7,952,008,248	56.5
Cobalt sulphate	5,556,108,180	39.5
Scandium oxide	375,000,000	2.7
Ammonium sulphate	183,319,200	1.3
Total	14,066,435,628	100

Table 1-13: Process design criteria summary

Process Plant Operating Assumptions	Units	Quantity
Autoclave Throughput	tpa	2,500,000
Operating hours per year	hours p.a.	7,665
Availability Leach Plant	%	87.5
Nickel HPAL Feed Design Grade	%	0.985
Cobalt HPAL Feed Design Grade	%	0.312
Scandium HPAL Feed Design Grade	ppm	53
Autoclave Operating Temperature	˚C	250
Autoclave Residence Time (minimum / nominal)	minutes	79 / 86
Sulphuric Acid to Leach	kg/t	240 - 290
Estimated Leach Extractions (Goethite)		
Nickel	%	95.7
Cobalt	%	95.0
Scandium	%	91
cRIP Metal Recovery (Ni/ Co & Sc)	%	99

Estimated Overall Recoveries (Goethite)		
Nickel	%	92.6
Cobalt	%	91.2
Scandium	%	31.5

1.6.2 Summary

Results of SRK's review of the 2018 Definitive Feasibility Study and supporting documentation, the QP's have concluded that the 2018 DFS summarises in this Technical Report contains adequate detail and information to support a Feasibility level analysis.

While no potential fatal flaws have been identified for the Project, the main areas of risk relate to project funding, execution systems, metallurgical assumptions and water availability.

With regards to the metallurgical assumptions, having reviewed Clean TeQ's DFS documentation and where appropriate other relevant information, and based on the information supplied, SRK concludes that:

- each of the individual flowsheet operations or steps selected in the overall process is technically and practically feasible
- the sequence of processing steps (the flowsheet) is technically and practically feasible
- the flowsheet selected is capable of producing the specified quality of products.
- the study has provided process and other engineering information such that all of the significant equipment selection and sizing can be undertaken

Clean TeQ have allowed for reduction in tonnes milled and metallurgical recoveries in the ramp-up period to mitigate the risk of achieving the predicted recoveries from the technical data. The assumptions were developed from data recovered when the various process steps were operating as well as could be expected at the pilot plant level. The metallurgical recoveries and plant throughput are presented in Table 1-14.

Table 1-14: Process plant ramp-up Physicals

	Units	Yr 1	Yr 2	Yr 3 onwards
Nickel recovery	%	75.9	88.4	92.6
Cobalt Recovery	%	75.1	87.5	91.2
Mill Feed	tpmth	117	204	208

1.7 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.

One of the Project's competitive advantages is its proximity to existing infrastructure. It is close to road and rail infrastructure which connects it to major sea freight corridors to international suppliers and to Asian markets.

The Project is located 30 km from a source of high voltage power from the electrical grid at Trundle. 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 operations workforce would be accommodated in the local communities.

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, workshops, stores, ablutions and change house, fences and security
- Construction camp
- IT and communications systems
- Sewage plant
- Store and laydown facilities
- Ore stockpiles and waste stockpile area
- ROM buffer 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.

1.7.1.1 Water Supply and Recycling

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.

River water will be used to supplement the bore water supply when there is abundant supply. Water will be recycled from the Tailings Storage Facility, and water loss will be minimised by using fin-fan cooler installations for the sulphuric acid plant, steam generators, crystallisers and for general plant cooling water.

1.7.1.2 Acid Production

Sulphuric acid is produced on site with a dedicated acid plant capable of making 2,700 t/day of sulphuric acid. Sulphur will be shipped to the Port of Newcastle as bulk cargo and will be containerised and then railed to a new rail siding, north of Trundle. Containers importing sulphur will be unloaded and stacked at the siding and the train will be loaded with a mix of empty containers and product containers for movement back to Newcastle. From the rail siding, trucks will shuttle standard 20ft sea containers to the plant.

1.7.1.3 Power and Steam Generation

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 de-superheated for distribution to the process plant users. In normal plant operation, all required heating is derived from the steam generated within the sulphuric acid plant. This provides heating to the pressure acid leach plant and high-pressure steam to a steam turbine generator installation. Lower pressure steam from the sulphuric acid plant is directed to the product crystallisers and dryers. A diesel-fired high-pressure steam boiler is provided to allow additional steam/power production when there is reduced sulphuric acid plant output.

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.

Most of the plant’s electricity requirements are supplied via the sulphuric acid plant cogeneration facility. Additional supplementary power is provided off a 66 kV tie-line from the New South Wales electricity supply grid. Several 2MW diesel engine powered electrical generators are installed for start-up scenarios and sustaining critical drives in power failure events.

1.7.1.4 Neutralisation

Limestone is required for neutralising the process slurries and liquors following acid leaching. There are a number of potentially viable sources of supply of Limestone in the region to The Project, two of which to date appear to be the most competitive contenders:

Westella

- The original FS obtained approvals to mine a maximum of 600,000 tpa of limestone from the Gillenbine limestone deposit, situated approximately 20 km southeast of the Project,
- The Limestone would be mined in Clean TeQ’s quarry and trucked to site. 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.

Ezylime

- Limestone could either be solely or additionally obtained from an existing local Limestone operation, Ezylime, of which a preliminary resource estimate and costing has been developed.

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.

1.7.1.5 Development Consent

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'. This includes in principal agreement of 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, and the limestone quarry.

1.8 Environment, permitting, social and community

1.8.1 Permitting

The Development Consent DA 374-11-00 has been modified on five occasions since it was issued in 2005 (Modifications 1, 2, 3, 5 and 6). A modification approved in 2016 (Mod 3) allows mining and processing operations 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 most recent modification to the Development Consent provided for changes to the accommodation facility at the Project which were deemed necessary to both optimise the mine plan and improve the amenity of the on-site workforce (Mod 6). The changes included the relocation of the accommodation facility from the main mine site to an adjacent property south of the mine on a property owned by the Company called "Sunrise".

A Development Consent modification to support several project optimising scope amendments (Mod 4) is well advanced with final approval currently anticipated in the second half of 2018. Mod 4 involves the implementation of these opportunities including mining in a more selective manner, addition of drilling and blasting, adoption of the resin-in-pulp processing method, increased sulphur demand and sulphuric acid production; increase limestone demand, addition of a cystalliser, changes to process input and road transport requirements, addition of a water treatment plant, increased tailings storage facility capacity, reduced evaporation pond capacity, relocation of mine infrastructure, addition of surface water extraction from the Lachlan River, minor changes to the borefield transfer station and reduced gas demand.

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.

Virtually all of the land required for project implementation is freehold land owned by Clean TeQ Sunrise. As a result, there is minimal risk of any future exposure to Native Title claims.

The company is in negotiations with the Crown and the State Government to agree access arrangements on the small amount of Crown land and state forest underlying the main Project area.

The company has signalled its intention to acquire additional freehold land, including properties required for a road to allow the primary logistics route to bypass the town of Fifield. At the time of writing, the company had agreed terms to acquire the land for the bypass road and was in the process of completing formal and binding documentation.

Easement agreements are proposed in order to secure land required for the electricity tie-line.

1.8.2 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 1-11: 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 remaining for the Project is the established borefield.

On 13 June 2006, the NSW Office of Water granted Water Bore licences to Clean TeQ for the extraction of 3,154 ML 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 in the past (for local use under a water purchase agreement).

SRK notes that the currently approved volume of water sourced from borefields is sufficient to meet the Project's entire raw demand. However, as a risk mitigation measure the Company's intention is to supplement with water taken directly from the Lachlan River. At the time of writing, Development Consent Modification 4, which contained a request to increase water supply security by including surface water from the Lachlan River, had been submitted, but approvals had not yet been obtained for this supplementary water source.

1.8.3 Community and stakeholder consultation

A community consultation program is being 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 an in-principle agreement for a voluntary planning agreement (VPA) 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.

Community consultation will continue throughout all stages of development.

1.9 Markets and Pricing

The key pricing assumptions were:

Nickel price	USD7/lb LME + USD1/lb sulphate premium
Cobalt price	USD30/lb LME/LMB + nil sulphate premium
Scandium oxide price	USD1,500/kg
Exchange Rate	USD0.75: AUD1.00

1.9.1 Nickel and Cobalt

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. Forecast annual demand growth for nickel sulphate for batteries is 22.5% compound annual growth rate to 2025 and 15.6% for cobalt. The lithium-ion battery market is primarily being driven by electric vehicles which are increasingly adopting cathode chemistries high in both nickel and cobalt because of their high energy density, long cycle life, safety and power output. These require high purity nickel and cobalt sulphates as precursor material for cathode production.

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)chemistries 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.

In its economic analysis, Clean TeQ has assumed a flat London Metal Exchange nickel price of USD15,430/t (USD7/lb) on a real basis for the life of the Project. Continued development of the electric vehicle industry is, in the long-term, likely to support higher nickel prices.

Clean TeQ has assumed that battery grade nickel sulphate produced at The Project will sell for a premium of USD2,205/t.

Clean TeQ has assumed a flat London Metal Exchange/London Metal Bulletin cobalt price of USD66,140/t (USD30/lb) on a real basis for the life of the mine. Clean TeQ's cobalt pricing assumption is based on the consensus forecast estimate for the long-term cobalt price of USD27.21/lb, published by a range of reputable brokers and banks who have active research into these commodities. A nil sulphate premium has been assumed.

1.9.2 Scandium

The Company believes that the scandium market has considerable latent demand potential however has historically suffered from significant supply constraints. While scandium oxide prices have historically ranged from USD2,000-4,000/kg (see US Geological Survey Commodity Reports), the DFS has assumed a forward price of USD1,500/kg, which is the price at which the Company expects significant additional demand growth to be stimulated.

1.10 Financial summary

The capital and operating costs for the processing options are summarised below.

1.10.1 Capital Estimate

All capital expenditures are shown in real terms over the construction period, have been estimated at a P50 (+/- 15%) level of accuracy, and are summarised in Table 1-15.

Table 1-15: Development Capital Expenditure

Major Area	Description	Cost AUD'000	Cost USD'000
	Direct Costs		
1000	Site Development	15,871	11,903
2000	Mining	23,179	17,384
3000	Ore Leach	330,778	248,083
4000	Refinery	198,404	148,803
5000	Reagents	184,050	138,037
6000	Services and Infrastructure	303,069	227,301
	Sub-Total Direct Costs (TDC)	1,055,351	791,513
8000	Indirect Costs: Including field Indirects, spares and first fills, commissioning & pre-commissioning, Alliance delivery	394,217	295,663
	Sub-Total Indirects	394,217	295,663
9000	Owner's Team (Including Owners Cost)	160,492	120,369
	Sub-Total Owner´s Costs	160,492	120,369
9000*	Estimate Contingency – P50	219,410	164,557
	Sub-Total Contingency & Risk	219,410	164,557
	Fee	158,318	118,739
	Sub-Total Fee	158,318	118,739
	TOTAL INSTALLED COSTS (TIC)	**1,987,788**	**1,490,841**

1.10.2 Life of Mine Operating Costs

The Project is unique among laterite projects due to its high cobalt content relative to nickel and its low acid-consuming elements such as calcium, magnesium and aluminium." The result is a Project that is expected to deliver lowest quartile C1 cash costs averaging negative USD-1.46/lb Ni once by-product credits from cobalt, scandium and ammonium sulphate are factored in.

The cobalt credits have a significant effect on C1 operating costs. The cobalt credits payable at the assumed cobalt price of USD30/lb result in an average USD5.60 credit for every pound of nickel produced.

All operating costs are expressed in real terms over the life of mine, post ramp-up (years 2-25).

Table 1-16: Estimated Annual Average Operating Cost (Years 2-25)

Area	Description	Estimate AUDM	Estimate USD/lb Ni	Estimate[2] USD/lb Ni - Co Credits
Mining	Owner's team labour	4.73	0.09	
	Crusher feed	1.18	0.02	
	Mining contractor	53.83	0.99	
	Grade control and assay	1.54	0.03	
	Drill and blast	0.83	0.02	
Sub-total for Mining		62.11	1.14	
Process Plant	Owner's team labour	45.90	0.84	
	Sulphur	57.30	1.05	
	Limestone	9.71	0.18	
	Ammonia	11.91	0.22	
	Quicklime	5.69	0.10	
	Other reagents	18.61	0.34	
	Maintenance cons. & shutdown lab.	10.39	0.19	
	Operating supplies	7.58	0.14	
	Power and water	14.20	0.26	
Sub-total for Process Plant		181.27	3.33	
General and Administration	Owner's team labour	5.86	0.11	
	Outbound logistics	3.85	0.07	
	Town administration	0.74	0.01	
	Insurance	1.00	0.02	
Sub-total for General and Administration		11.45	0.21	
Total		**254.83**	**4.68**	**(5.60)**

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[2] Excludes scandium and ammonium sulphate by-product credits

Table 1-17: Operating Cost Assumptions (Years 2-25)

Life of Mine Costs (Post Ramp-Up Years 2 to 25)	USD/lb Nickel (Real)		USD/lb Nickel (Real) Net of Cobalt Credits	
	C1	Total Cash Costs FOB	C1	Total Cash Costs FOB
Mining costs	1.14	1.14	1.14	1.14
Processing costs	3.33	3.33	3.33	3.33
General, administration and other site overheads	0.14	0.14	0.14	0.14
Haulage and port	0.07	0.07	0.07	0.07
By-product credits	-	-	-	-
Cobalt credits	-	-	-5.60	-5.60
Scandium oxide credits	-	-	-0.37	-0.37
Ammonium sulphate credits	-	-	-0.18	-0.18
Cost Subtotal (C1)	**4.68**	**4.68**	**-1.46**	**-1.46**
Nickel and cobalt royalty	-	0.41	-	0.41
Ivanhoe royalty	-	0.34	-	0.34
FOB cash costs	-	**5.43**	-	**-0.71**

1.10.2.1 Owners Reserve

Provision for an additional owner’s reserve of USD106.5M is recommended to allow for the following items not included within the capital development cost:

- additional contingency provision required to increase the probability of delivering the project within the estimated capital cost from 50% (P50) to 75% (P75) = (USD76.7M)
- escalation provision amounting to 2% of the total cost = (USD29.8M).

1.10.2.2 Sustaining Capital and Closure Costs

Provision had been made within the financial analysis to allow for the following sustaining capital and mine closure costs as follows:

- Tailings Storage Facility and sustaining capital (USD232.9M)
- closure and decommissioning costs (USD100.1M).

1.10.3 Project valuation

The summary of financial modelling post-tax outputs is given in the Table 1-18.

Table 1-18: Valuation Summary of Outcomes (Post Tax)

Parameter	Units	Assumption /Outcome
Autoclave Throughput	Mtpa	2.5
Initial Life of Mine	Years	25
Initial Life of Mine Revenue	USD Bn	14.07
Initial Life of Mine EBITDA	USD M	8,600
Average Annual EBITDA	USD M	344
Pre-production Capital Estimate	USD Bn	1.49
Net Present Value (NPV)	USD M	1,392
Internal Rate of Return (IRR) (post tax)	%	19.1
Average C1 operating cash costs (Year 2-25) – inclusive of by-product credits	USD/lb Ni	-1.46/lb
Average C1 operating cash costs (Year 2-25) – exclusive of by-product credits	USD/lb Ni	4.68/lb
Project payback (simple)	Years	4.3

1.10.4 Discounted Cash Flow Analysis

Cash flow modelling of the Project, summarised in Table 1-19 at 100% equity demonstrates an NPV_8 of USD1,392M (at a post-tax, real discount rate of 8%) at an Internal Rate of Return (IRR) of 19.1% (30% corporate tax rate reducing to 25% over the duration of the Project), in line with proposed Enterprise Tax Plan No. 2 201710, with total earnings before interest, tax, depreciation and amortisation (EBITDA) cash flows over the 25 year project life of USD8,600M.

NPV calculations exclude pre-FID expenditure totalling USD173M (definitive feasibility study and development capital prior to December 2018).

Commodity prices were assumed to be a flat USD8/lb for nickel (inclusive of a USD1.00/lb sulphate premium) and USD30/lb for cobalt for the life of mine.

Average annual EBITDA over the life of mine is estimated to be USD344M, peaking at ~USD550M in years when the operation is running at maximum production rates.

Table 1-19: Valuation Summary Pre-Tax Net Present Value

Key Metrics	Unit	Value
Post-tax NPV at 31 December 2018, 8% (real)	USDM	1,392
Post-tax IRR (real)	%	19.11
Simple payback (years, real CF) from 01 Jan 21	Years	4.33

1.11 Risks and opportunities

Results of SRK's review of the 2018 Definitive Feasibility Study and supporting documentation, the QP's have concluded that the 2018 DFS summarises in this Technical Report contains adequate detail and information to support a Feasibility level analysis.

While no potential fatal flaws have been identified for the Project, the main areas of risk relate to project funding, execution systems and metallurgical assumptions.

The primary environmental consent required for project implementation has been granted.

SRK has no reason to expect that the secondary approvals required for the project would be refused.

The planned waste dump capacity of approximately 120 Mt is insufficient for the total waste production of 165 Mt. The 120 Mt capacity will be exceeded in year 15 of the project. Similarly, the planned medium to low grade ore stockpile capacity available is approximately 30 Mt, peaking at 93 Mt. The 30 Mt capacity will be exceeded in year 11 of the project.

Various options exist to increase the available capacity within ten years of project start-up and Clean TeQ is confident that one, or a combination, of the options will be successfully negotiated to solve this lack of capacity within the ten-year timeframe.

With regards to the metallurgical assumptions, having reviewed the 2018 DFS documentation and where appropriate other relevant information, and based on the information supplied, SRK concludes that:

- each of the individual flowsheet operations or steps selected in the overall process is technically and practically feasible
- the sequence of processing steps (the flowsheet) is technically and practically feasible
- the flowsheet selected is capable of producing the specified quality of products
- the study has provided process and other engineering information such that all of the significant equipment selection and sizing can be undertaken.

Clean TeQ have allowed for reduction in tonnes milled and metallurgical recoveries in the ramp-up period to mitigate the risk of achieving the predicted recoveries from the technical data. The assumptions were developed from data recovered when the various process steps were operating as well as could be expected at the pilot plant level. The metallurgical recoveries and plant throughput are presented in Table 1-20.

Table 1-20: Process plant ramp-up Physicals

	Units	Yr 1	Yr 2	Yr 3 onwards
Nickel recovery	%	75.9	88.4	92.6
Cobalt Recovery	%	75.1	87.5	91.2
Mill Feed	tpmth	117	204	208

1.11.1 Risks

The DFS included a risk assessment process consistent with ISO 31000:2009 Risk Management.

Attendees included:

- Clean TeQ personnel
- SNC-Lavalin Australia Pty Ltd personnel
- project execution personnel
- operations/maintenance personnel
- independent technical personnel
- mining consultant personnel.

The register contains, for each identified risk:

- risk assessment
- agreed risk treatment (mitigation) plan (where required)

- an individual assigned responsibility for ensuring the treatment plan is implemented
- specifies a due date for completion (where applicable).

Figure 1-12 and Figure 1-13 show both the risk profile at completion of the DFS, and the mitigated risk profile (i.e. anticipated profile at financial investment decision).

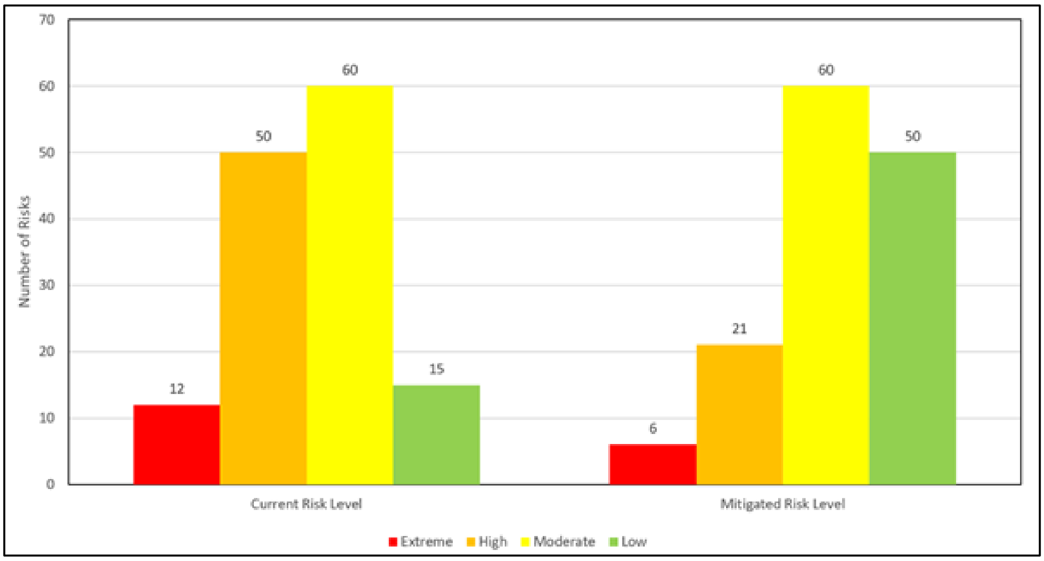
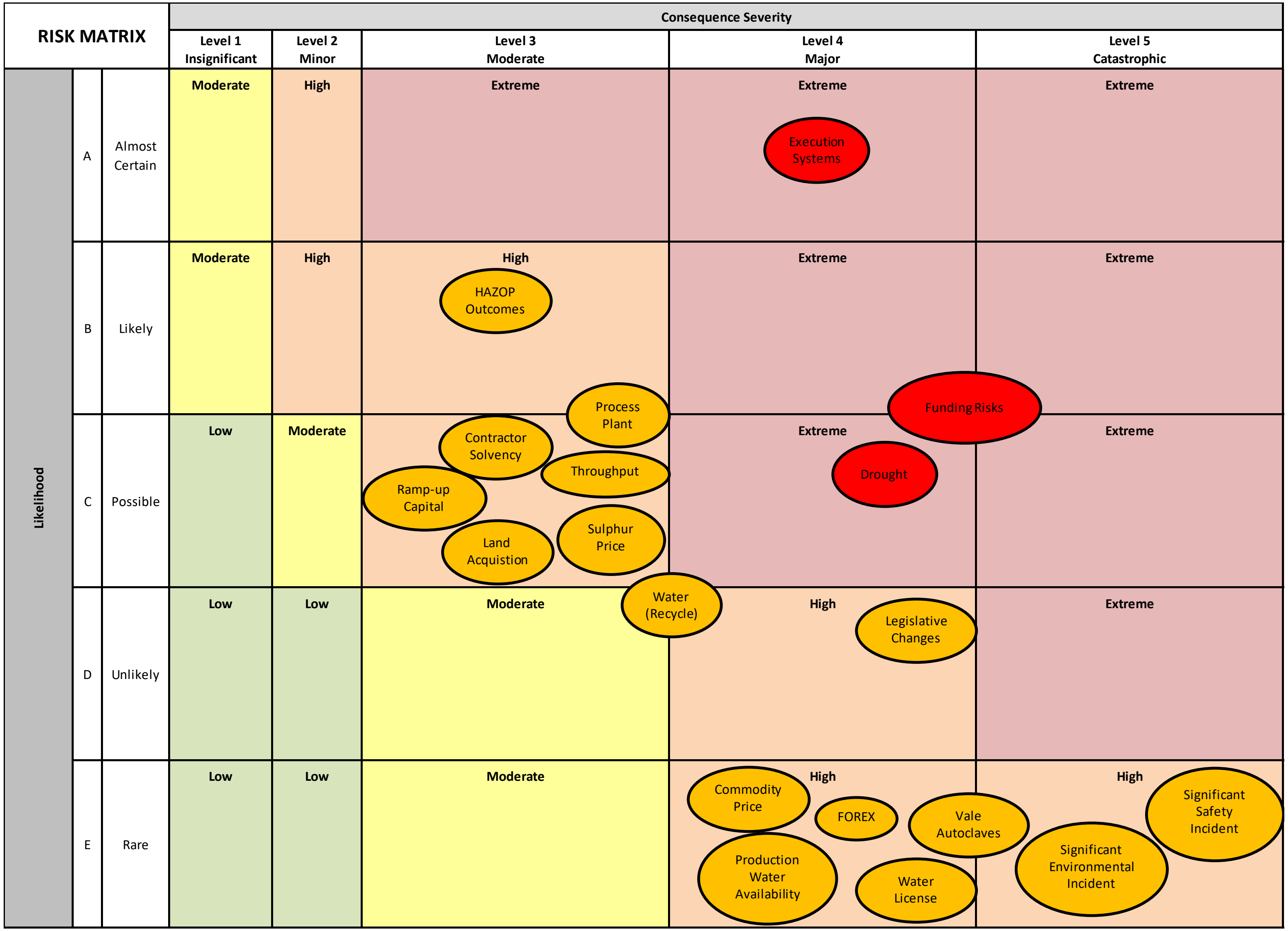


Figure 1-12: Risk Reduction – from DFS to Financial Investment Decision



RISK MATRIX			Consequence Severity				
			Level 1 Insignificant	Level 2 Minor	Level 3 Moderate	Level 4 Major	Level 5 Catastrophic
Likelihood	A	Almost Certain	Moderate	High	Extreme	Extreme	Extreme
	B	Likely	Moderate	High	High	Extreme	Extreme
	C	Possible	Low	Moderate		Extreme	Extreme
	D	Unlikely	Low	Low	Moderate	High	Extreme
	E	Rare	Low	Low	Moderate	High	High

Figure 1-13: Clean TeQ Sunrise – Mitigated Risk Profile Bubble Diagram

1.11.2 Opportunities

A number of opportunities were identified in the DFS with the potential to improve the operability, performance and output of the processing plant, Table 1-21.

Table 1-21: Project risks and opportunities

Risk/ Opportunity	Summary
Technology	• The HPAL process is now in its fourth generation, with many of the learnings of previous operations incorporated in revised HPAL design work. • cRIP technology has successfully been implemented since the 1950s on over 30 full-scale operations to recover a range of metals. Clean TeQ has spent over 13 years developing and piloting the technology on laterite ores for optimal recovery of nickel, cobalt and scandium. • The resource and mining risk relating to the Project is assessed to be low because of the continuity of the deposit and the use of well-established resource estimation techniques. The reliability of nickel laterite resource and mining reserve estimates has improved with experience.
Political	• The primary Development Consent required for project implementation was granted in May 2017 and included permitting of water source and use (whether surface water or groundwater) to satisfy operational water requirements. • 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. • 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. • Mining Leases under NSW Mining Act 1992 for the main project area and the limestone quarry have been granted and are in good standing. • With a granted Development Consent, development of the Project is subject only to financing. The Project site is in an area of no particular environmental significance and the Project has the support of Local and State Government leaders. • 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. • Land access/ acquisition agreements need to be finalised to cover parts of the Project area, limestone quarry, and rail siding. • Additionally, as a risk-mitigation measure, it would be desirable for the current water allocation to be increased, or additional licences purchased. • 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. Currently, cobalt supply is largely from Africa and nickel laterite developments tend to fall off rapidly in cobalt production capacity within the first five years of operation. To date, Clean TeQ Sunrise has secured offtake agreements for 20% of its Ni-Co production.

Risk/ Opportunity	Summary
Implementation	• Implementation issues include ensuring that the required technical standards in design and construction are met 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. • 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: - recruiting and training the required numbers of suitably qualified and experienced management, supervision and operations personnel - establishing a suitable industrial relations structure for operations staff • The workforce establishment strategy emphasises recruitment of an experienced operations management team and to exploit knowledge and experience gained from currently operating lateritic nickel operations. Early operator and maintenance personnel training programs are to be adopted for local employees, and more experienced personnel from similar operations.

2 Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) has worked in conjunction with Widenbar & Associates (Widenbar) and Longley Mining Consultants Pty Ltd (LMC) and Clean TeQ Holdings Limited to prepare this Technical Report on the Sunrise 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.

Clean TeQ Holdings Limited 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 in the 2018 DFS on the Project to sufficient detail to enable SRK, Widenbar and LMC to act as Qualified Persons and preparing documentation as required for the NI 43-101 compliant Technical Report on the Sunrise 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 during the months of May to July 2018.

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 and Longley Mining Consultants Pty Ltd (LMC) and Clean TeQ Holdings Limited (the Owner) on the Sunrise 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 owner’s 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.

The 2017 Mineral Resource Estimation was completed by Rob Longley, Principal Consultant (Longley Mining Consultants Pty Ltd), BSc(Geology)(Hons), MAIG. By virtue of his education, membership of a recognised professional association and relevant work experience, Rob Longley is an independent Qualified Person (QP) as this term is defined by the 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.

James Kyle, Principal Metallurgist, B.Sc.(Hons), Ph.D.(Chemistry), Postgrad Dip. (Extract. Met.), MBA, FAusIMM (No 106344), 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, James Kyle is an independent QP as this term is defined by NI 43-101.

Scott McEwing, Principal Consultant (Mining), BENG (Mining), FAusIMM (No. 111809), CP (Mining) reviewed the mining and Mineral reserve aspects of the project. By virtue of his 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 25 June 2018 is based on the 2018 DFS 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/ LMC 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.

Table 2-1: List of qualified persons

Qualified Person	Employer	Last site visit date	Professional designations	Area of responsibility and Report sections
Peter Fairfield	SRK	07/06/17	BEng (Mining), FAusIMM CP (Mining)	Sections 1 - 5, Sections 19 - 27
Lynn Widenbar	Widenbar & Associates	21/09/17	BSc (Geology), MSc (Mineral Exploration) DIC MAusIMM MAIG	Section 14
Rob Longley	Longley Mining Consultants Pty Ltd	07/04/18	BSc(Geology) (Hons) MAIG	Section 6-12
Scott McEwing	SRK	NA	BEng (Mining), FAusIMM CP (Mining)	Sections 15 - 16,
James Kyle	SRK	NA	B.Sc.(Hons), Ph.D.(Chemistry), Postgrad Dip. (Extract. Met.), MBA, FAusIMM,	Sections 13,17,18

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 Project (previously known as Syerston), is situated in central New South Wales, about 440 km west-northwest of Sydney. The Project is well supported by major centres, with the mining communities of Parkes and Condobolin located within 100 km of the Project area and Dubbo located approximately 160 km to the northeast (Figure 4-1). The town of Fifield is located 4 km from the Project area. The Project area has an average annual rainfall of 458 mm, high daytime temperatures in summer and low minimum temperatures in winter.

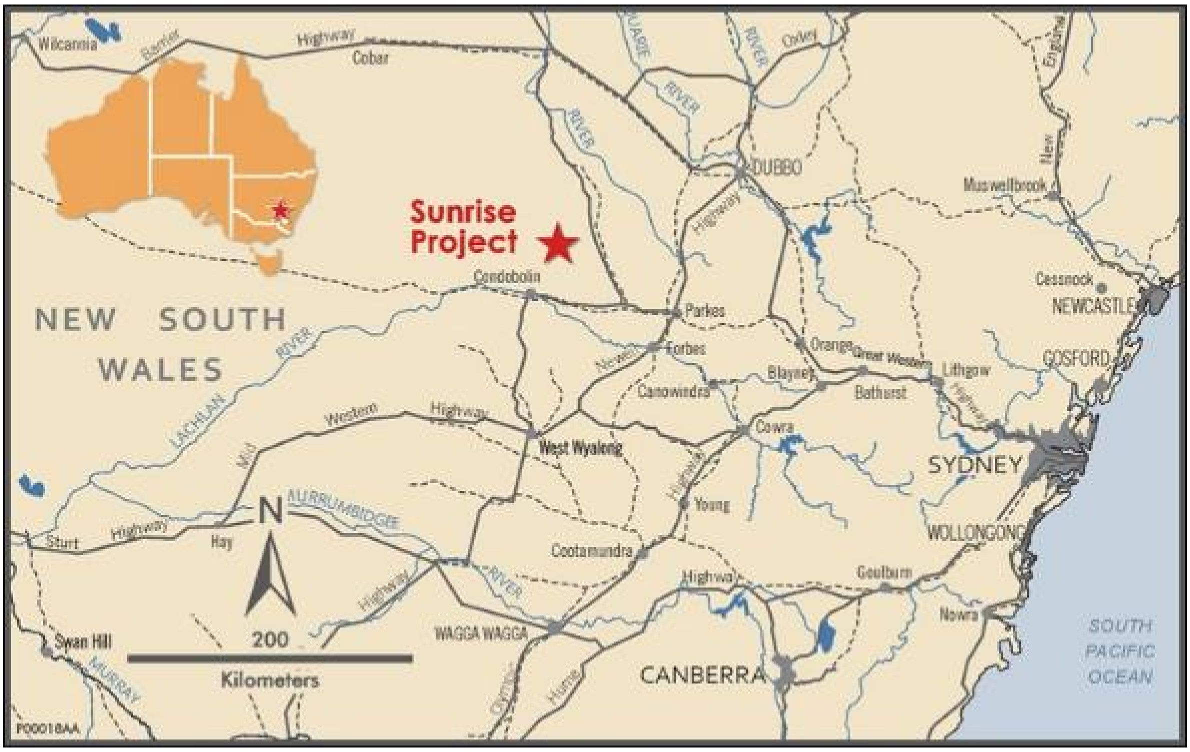


Figure 4-1: Project Location Map

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 north-eastern corner of the Project area.

One of the Project's competitive advantages is its proximity to existing infrastructure. The Project is located ~30 km from a source of high voltage power from the electrical grid, a rail line passes within 25 km of Clean TeQ Sunrise and bitumen roads provide good access to the site. Major centres in the area (Parkes and Dubbo) 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 in the shires of Lachlan and Parkes, while the bore field supplying water for the Project is located further to the south in the Forbes Shire.

4.2 Landholding

Figure 4-2 and Figure 4-3 show the landholding over the Project area and the Westella limestone deposit. These properties are Syerston, Sunrise, Kingsdale, Slapdown and Westella. Clean TeQ Sunrise Pty Ltd owns the land which is the subject of the rail siding.

Negotiations for new options, acquisitions or access/easement agreements will need to be undertaken for the Fifield bypass

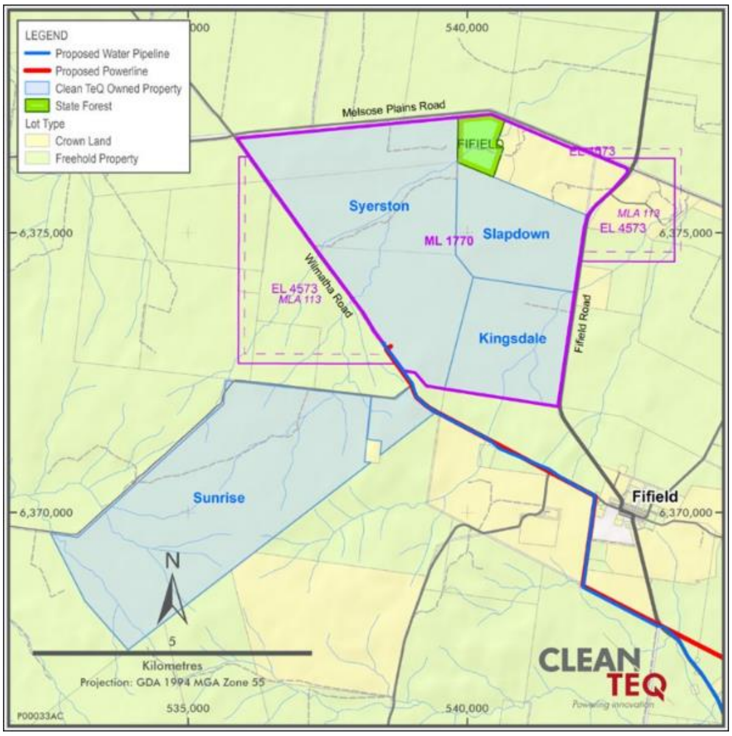


Figure 4-2: Clean TeQ Sunrise Pty Ltd Landholding - Main Project Area

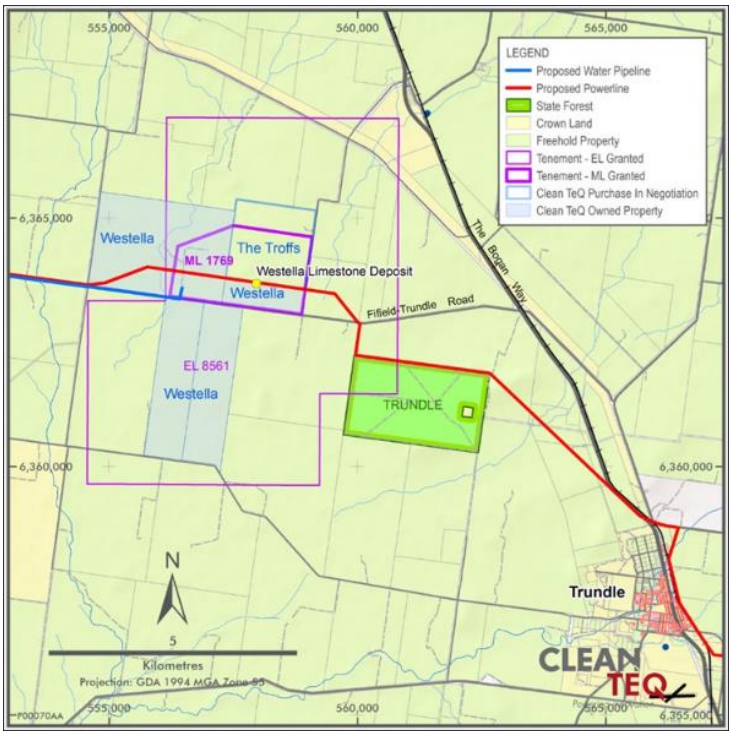


Figure 4-3: Clean TeQ Sunrise Pty Ltd Landholding - Limestone Quarry

4.3 Mining tenure

4.3.1 Exploration Licences - EL4573 and 8561

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 Licences over the area of the Licence.

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

The Exploration License (EL4573) underlying the area for the Project and facilities is owned by Clean TeQ Sunrise Pty Ltd. The status of the Exploration License as of June 2018 is listed in Table 4-1.

Table 4-1: Exploration Licence Details

Licence Number	Date of Grant	Expiry Date	Area Granted	Current Area (ha)	Ownership (%)	Annual Expenditure (AUD)
EL4573	17 Aug 1993	16 Aug 2018 Renewed 3rd August 2018	18 units	3856	100	50,000
EL8561	11 May 2017	11 May 2020	13 units	3744	100	25,000

EL4573 and EL8561 have been granted subject to the provisions of the NSW Mining Act 1992 and to the conditions of each licence. Under EL4573, Clean TeQ has exclusive rights to prospect in the exploration area for Group 1 minerals.

Group 1 minerals are elemental minerals (metallic) which include rare earth minerals (including scandium), nickel, cobalt and platinum. Under EL 8561, Clean TeQ has exclusive rights to prospect in the exploration area for Group 2 minerals (non-metallic), which include limestone.

At the expiry dates Clean TeQ has the option to renew the Exploration Licences for an additional three years, subject to the licence conditions being met. To date, all Exploration Licence conditions for EL4573 and EL8561 have been met.

4.3.2 Mining Leases and Mining Lease Applications – ML1769, ML1770 and MLA113

The boundaries of the granted Mining Leases and pending Mining Lease Applications which have been applied for over the Project area and Westella limestone deposit are shown in Figure 4-2 and Figure 4-3.

The status of each Mining Lease and Mining Lease Application is described in Table 4-2.

Table 4-2: Mining Lease Applications

Licence Number	Area (ha)	Application/ Grant Date	Grant	Expiry Date	Interest (%)
ML 1769	389.7	15 February 2018	Granted	15 February 2039	100
ML 1770	2,195.0	16 February 2018	Granted	16 February 2039	100
ML A113	735.6	10 August 1998	Pending	Pending	100

ML 1770 covers the main Project Area which includes the mine, the processing facilities and associated infrastructure. The area covered by the remaining section of MLA 113, which was not included in ML 1770, is to the west and outside the main project area. When granted, it will be a separate ML with different conditions restricting it’s use and purpose to ancillary activities only. ML 1769 and ML 1770 were granted in accordance with the NSW Mining Act 1992 and Regulations (the Act).

The Act administered by the Department of Mineral Resources, allows the grant of a Mining Lease for any mineral or minerals defined under the Act. It also prescribes a royalty rate payable on all minerals recovered from the Mining Lease.

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.

Development consent for The Project is held by Clean TeQ Sunrise Pty Ltd. The Project has progressed through several Modifications since the original Development Consent. Modification 3 and subsequent Modification applications have occurred under Clean TeQ’s ownership of the Project. Summary and detailed description of the currently approved Development Consent Modifications and Modification Applications are described in detail in Section 20.

With commencement of the NSW Environmental Planning and Assessment Amendment Act 2017 on 1 March 2018, Schedule 6A of the Environmental Planning and Assessment Act 1979 (EP&A Act) was repealed and the provisions of Schedule 6A were transferred to a new regulation under the EP&A Act, which ended the Part 3A transitional arrangements and resulted in the Project becoming a State significant development under the EP&A Act. Any future modification applications of the Project will have to comply with relevant State significant development legislative requirements.

Mining Leases under NSW Mining Act 1992 for the main project area and the limestone quarry have been granted. Having completed an EIS, having a Development Consent and Mining Leases in place offers a significant advantage over other projects, due to the time and relative costs to complete these activities. The previous granting of a Development Consent indicates that there are unlikely to be any insurmountable environmental obstacles.

A number of important secondary approvals must be granted before the project can be implemented. Details of additional permits and licences required are discussed in Section 20. 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 will not be granted

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.

4.5 Liabilities

A payment of AUD43,4700 has been paid as part of the Mining Operations Plan. There are no known environmental liabilities.

4.6 Royalties

Royalties included in cost assumptions include both a state government royalty and a royalty to Ivanhoe Mines.

The New South Wales State Government (State Government) royalty rate for nickel, cobalt and scandium is 4% ex-mine value (less allowable deductions). This has been calculated by deducting processing and administration costs from revenue, as well as deducting capital allowances and other specific deductions. The royalty rate is then applied to the net revenue calculated.

Details provided by the NSW Government are available from:

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

The Royalty to Ivanhoe Mines is calculated with a 2.5% royalty rate applied to revenue net of the NSW State Government royalty.

4.7 Taxes

4.7.1 General

Clean TeQ falls under the Australian taxation system as an incorporated Australian legal entity, with the Project taxed as a separate legal entity.

The Project uses Australian tax rules to estimate tax payable:

- capital costs and certain expenses are capitalised during construction as assets and depreciated for tax purposes on a diminishing basis (diminishing value method) using a tax asset life of 15 years
- all expenses incurred during operations are assumed to be tax deductible, with capital costs depreciable
- The Clean TeQ group of companies has tax losses available which it will use over the first profitable years of the project
- Clean TeQ Sunrise Pty Ltd is registered for Australian Goods and Services Tax (GST), hence GST has been excluded from all cost and revenue estimates, including capital and operating costs. No GST cashflow impacts have been included in the analysis outlined in this chapter
- corporate tax rate is 30% and assumed to fall to 25% over the duration of the Project, applied to profits after allowances for all cash operating expenses, royalties, and capital depreciation is applied. These assumptions have been made based on the proposed Enterprise Tax Plan No. 2, 2017 timeframes and proposed changes
- any tax payable in any quarter has been assumed to be paid at the end of the quarter.

4.7.2 Excluded Cost Items

As identified above, the Project has been valued as a stand-alone asset thus there are some costs that Clean TeQ will incur as part of their normal business operation which have not been included in the valuation of the Project.

The corporate insurance & corporate legal costs were included in the capital development value but have been excluded from ongoing operating costs. The following corporate overhead costs required to manage the business exclusive of the project were not included in recurrent operating costs:

- non-site-based training
- salaries for company directors and senior management
- miscellaneous costs associated with maintaining an off-site head office
- revenue from other assets held by the company
- existing liabilities

- existing employee entitlements
- vendor loans
- future cost of tenement exploration.

4.8 Summary and conclusions

The primary environmental consent required and main mining leases for project implementation have 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.

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 11,000) located 85 km to the southeast; Dubbo (population 39,000) located 160 km to the northeast; and Condobolin (population 3,700) located 60 km to the southwest. Other small towns near the project include Tullamore (population 420), Tottenham (population 450), Trundle (population 680) and Fifield (population 280).

The Project is 4 km to the northwest 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 1980’s.

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 north-eastern 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 north-eastern quarter of the lease area contains land that has been subject to historical open cut mining for magnesite. 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, with tributaries in the southwest (see Figure 5-1). Several low gravelly hills occur across the site with broad shallow valleys between.

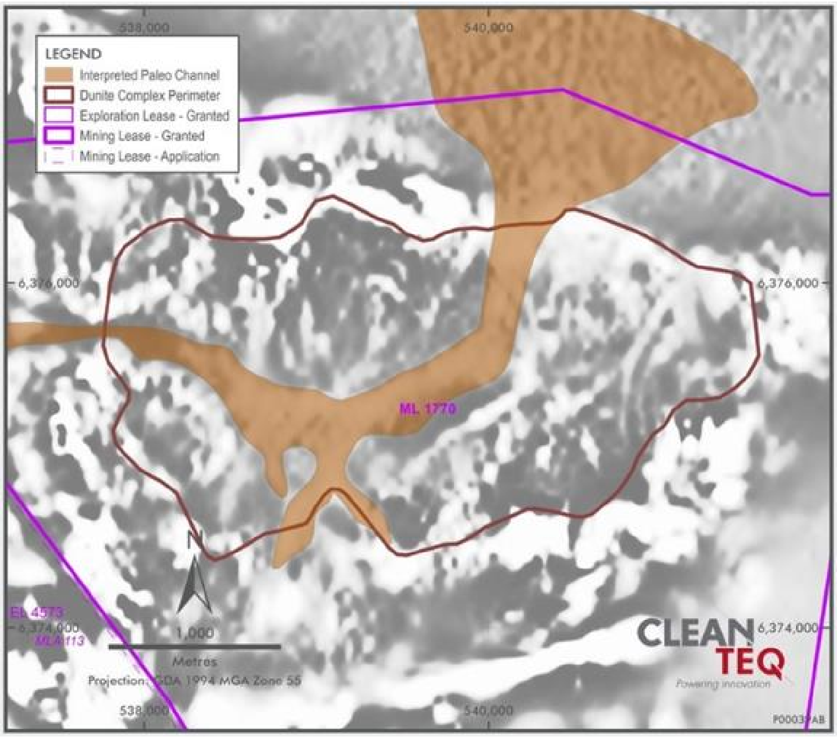


Figure 5-1: Main Paleo-Channels

5.4 Climate

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 458mm (as shown in Table 5-1 and Figure 5-2). 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 from the Condobolin Agricultural Research Station

Month	Mean Maximum Temperature (°C)	Mean Minimum Temperature (°C)	Mean Rainfall (mm)
January	33.8	18.2	46.9
February	32.8	18.2	44.8
March	29.4	14.8	39.6
April	24.6	9.9	30.6
May	19.6	6.5	35.7
June	15.7	3.9	31.3
July	15	2.8	35.9
August	16.9	3.5	33.7
September	20.6	5.8	31.4
October	25	9.4	47.4
November	28.8	13.1	38.6
December	31.7	15.7	41.6
Annual Average	24.5	10.2	457.8

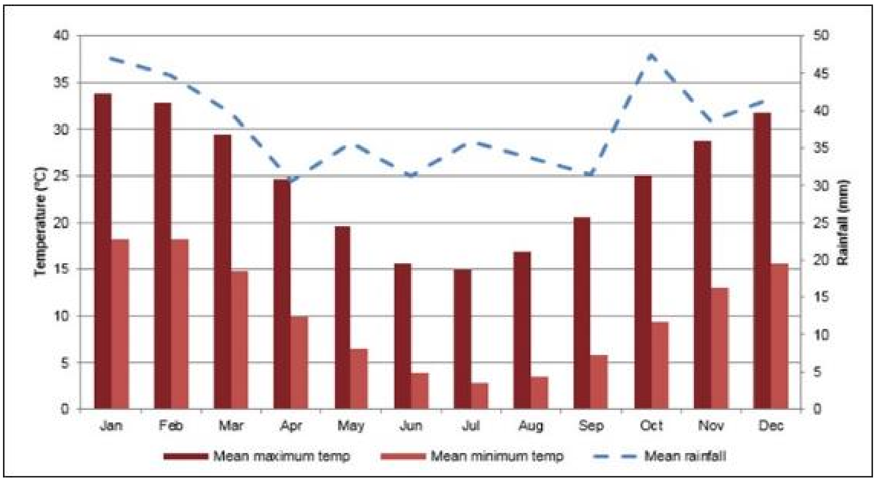


Figure 5-2: Climate Data from the Condobolin Agricultural Research Station

5.5 Infrastructure and local resources

The Project's competitive advantage is based in part on its proximity to existing infrastructure (see Figure 5-3). A rail line lies within 25 km of the Project, giving access to the ports of Sydney and Newcastle. Bitumen roads provide good access to the site.

Parkes and Dubbo have excellent infrastructure, including transport, airport and rail facilities, all of which are available to service the requirements of the Project. Condobolin also has rail facilities and a registered aerodrome. In addition, the mining industry in the region is well understood and supported within the major centres of Parkes and Dubbo. The Project and associated infrastructure are located within the Lachlan and Parkes Shires, with the bore field located in the Forbes Shire. All parties have responded positively to the Project.

There are several large operating mines in the region, including the North Parkes, Peak Hill, Browns Creek, Lake Cowal and Cadia mines.

Mining activity in the region supports the economies of the local towns. The former magnesite mine, which forms part of the Project area, employed up to 60 local residents.

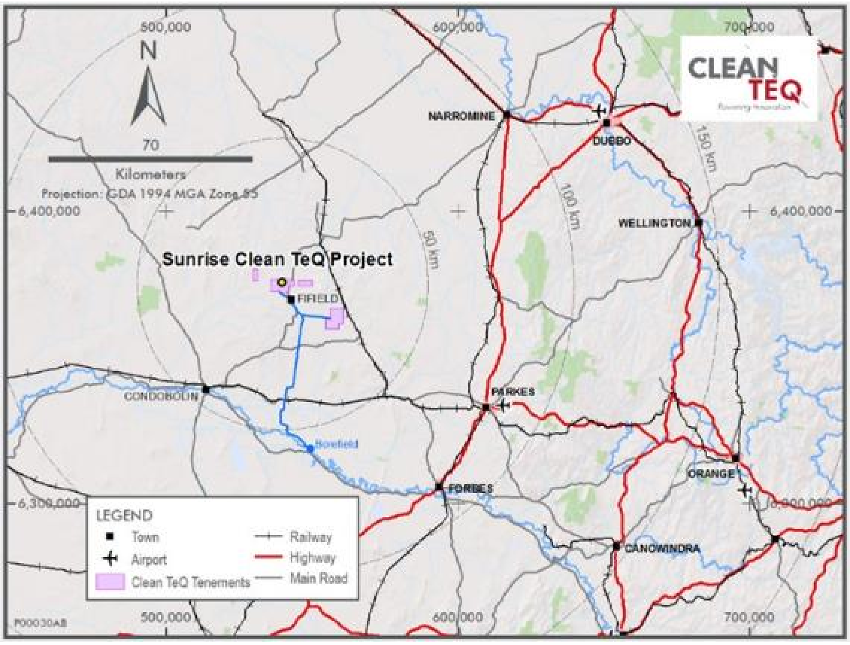


Figure 5-3: Regional Location Plan

6 History

The Fifield district is the only recorded source of platinum production in Australia. Reported production from the district totalled 639.5kg of platinum between 1887 and the mid 1960's. 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 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 in the Fifield area until March 1988, mainly searching for primary platinum within the Tout Complex (see Figure 7-2). The exploration failed to locate or identify any prospective platinum targets, although it did identify the occurrence of a significant laterite profile, which was enriched with nickel, cobalt and platinum, lying over the dunite portion of the intrusive.

During 1988, Poseidon Limited (Poseidon) entered into a joint venture agreement with Noble in this area. Poseidon was entitled to earn 51% of the project by spending AUD1.2M 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 joint venture, 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 core 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/AC drilling and metallurgical investigations. Noble's original ELs, 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 RC holes and three PQ triple-tube diamond core holes. This led to the resource being remodelled and estimated by EMC Consultants, which formed the basis of a pre-feasibility study. During this time, the Company entered into a joint venture agreement and option deed with Uranium Australia, later renamed Black Range Minerals (Black Range), to provide further funding for the Project.

Commencing in March 1998, Fluor Daniel Pty Ltd conducted a PFS on the nickel/cobalt resource and mining, processing and infrastructure facilities required for the development of the deposit. The Fluor Daniel PFS, which 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

Once Black Range Minerals took control of the project, they 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, several 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 Australia Pty Ltd (SLA) was commissioned by Black Range in June 1999 to undertake a feasibility study for the Project, which was completed in June 2000 (SLA, 2000[3]).

Based on the feasibility study, Black Range completed an Environmental Impact Statement (EIS) as a basis for its Development Consent application. The Environmental Impact Statement was submitted to the New South Wales 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 Limited. They 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 SLA to update the feasibility study carried out 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 a pressure acid leach environment with a 2.5 Mtpa autoclave feed capacity and mean recoveries of 93%. The processing plant had an output capacity of 25,000t of nickel metal and 5,000t 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 (AUD1.0B) with operating costs running at USD2.68/lb nickel, after cobalt credits. At the 2005 base case prices of USD4.10/lb nickel and USD12.00/lb cobalt, the project generated an internal rate of return 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 Sunrise project.

In 2016, Clean TeQ prepared a PFS as an update to the 2005 feasibility study (Clean TeQ, 2016).

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[3] SNCL, 2000. Syerston Nickel Cobalt Project, Feasibility Study, Volume 3B, Geology and Resource Estimate. Report to Black Range Minerals NL, July 2000 [Unpublished].

The 2016 PFS was based on a 2012 JORC Code compliant, Measured, Indicated and Inferred resource, completed by McDonald Speijers Pty Ltd (McDonald Speijers) in 2005 (McDonald Speijers, 2005a[4] and 2005b[5]). The resource estimate was based on more than 1,200 historical drill holes. 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.

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[4] McDonald Speijers, 2005a. Ivanplats Syerston Pty Limited, Syerston Project. Volume 1 Geology and MRE, 31 May 2005 [Unpublished].

[5] McDonald Speijers, 2005b. Ivanplats Syerston Pty Limited. Syerston Nickel Cobalt Project. Feasibility Study Update September 2005. Volume 3 Geology. [Unpublished].

Table 6-1: Summary of Drilling Programs up to November 2015

Time Period	Operator	Hole Type	Total			Used in 2017 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 platinum 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 and 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 and 22/03/99) and lacking details of the drilling contractor. However, co-ordinates are not the same as matching FR series numbers.
		RAB	P30 and 36	2	73	P30 and 36	0	0	Start and end dates and drilling contractor as above. No assay data. Same depths and within 20m in co-ordinates of FR30 and 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 platinum 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.
		RC	SRC001-340	341	14,149	SRC001-340	339	14,069	Infill drilling

Time Period	Operator	Hole Type	Total			Used in 2017 Resource Model			Comments
			Hole Number Range	Number of Holes	Metres in Database	Hole Number Range	Number of Holes	Metres in Database	
Aug 1997 - Aug 1998	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	25,870	SRC341-1076	724	25,710	Infill drilling; additional holes drilled after August 1999 are not reported in feasibility study documents. Note: SRC1061 to SRC1064 are excluded.
		DDH	SDD6-13	8	319.1	SDD6-13	0	0.0	PQ3 core for density and metallurgy tests; holes SDD6,8,9,10 and 12 were not assayed.
		Calweld	SCW1-9	9	234	SCW1-9	0	0	Large diameter holes; three holes in November 1999 were not reported in the feasibility study documents.
Feb 2005 - Mar 2005	Ivanplats Syerston	RC	SRC1077-1193	117	4,325	SRC1077-1193	117	4,325	Local 30x30m infill drilling.
			SRC1194-1225	32	1,594	SRC1194-1225	32	1,594	Infill drilling around elevated platinum intercepts.
			SRC1226-1251	26	833		0	0	Twinning previous Calweld and RC holes.
Aug 2014	Ivanplats Syerston	RC	SRC1263-1276	14	381	SRC1263-1276	14	381	Testing for peripheral scandium zones.
Apr 2015	Scandium21	RC	SRC1277-1310	34	944	SRC1277-1310	34	944	Infill drilling in areas of elevated scandium.
Nov 2015	Scandium21	RC	SRC1311-1368	58	1436	SRC1311-1368	58	1436	Infill drilling in areas of elevated scandium.
Project Totals		**RAB**		**207**	**6,891**	**-**	**0**	**0**	-
		Aircore		**148**	**4,275**	**-**	**45**	**1,441**	-
		RC		**1,354**	**49,532**	**-**	**1,318**	**48.459**	-
		DDH		**15**	**820**	**-**	**0**	**0**	-
		Calweld		**9**	**234**	**-**	**0**	**0**	-
		ALL		**1,733**	**61,752**	**-**	**1,363**	**49,900**	-

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 nickel equivalent (NiEq) cut-off grades.

The 1999 resource estimate was reported under the JORC Code and the 2005 resource estimate was reported under the NI 43-101 reporting guidelines.

In 2016 resource estimation for nickel and cobalt was undertaken by McDonald Speijers. scandium grades were independently estimated by OreWin Pty Ltd.

The 2016 resource estimate undertaken by McDonald Speijers was reported to the Australian Market under the guidelines of the JORC Code (2012) on 22 August 2016. (Table 6-4).

The historical estimates referenced above (199, 2005 and 2016) are not considered as current estimates and have no relevance to the current Project. The current estimate is the 2017 Widenbar estimate, released 7 October 2017 under the JORC Code (2012)

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

* 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.

Table 6-4: 2016 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.

Table 6-5 provides a summary of the current resource estimate at a range of different nickel equivalent cut-off grades, including selected accessory elements.

Table 6-5: 2016 Mineral Resource Estimate at a range of NiEQ cut-off[6] grades

Cut-off NiEQ %	Classification Category	Tonnage (Mt)	***Grades***								
			Ni %	Co %	***Accessory Elements***						
					Pt g/t	***Sc* ppm***	***Fe %***	***Al %***	***Ca %***	***Mg %***	***Mn %***
0.6	Measured	52	0.73	0.11	*0.20*	*51*	*35*	*2.5*	*0.38*	*1.3*	*0.80*
0.6	Indicated	49	0.58	0.10	*0.21*	*56*	*35*	*2.3*	*0.40*	*1.2*	*0.70*
0.6	**Meas + Ind**	**101**	**0.65**	**0.10**	***0.21***	***54***	***35***	***2.4***	***0.39***	***1.2***	***0.75***
0.6	Inferred	8	0.54	0.10	*0.16*	*78*	*36*	*2.8*	*0.38*	*1.4*	*0.78*
0.8	Measured	37	0.81	0.13	*0.21*	*53*	*37*	*2.5*	*0.28*	*1.1*	*0.93*
0.8	Indicated	26	0.66	0.13	*0.23*	*53*	*37*	*2.3*	*0.30*	*1.0*	*0.87*
0.8	**Meas + Ind**	**64**	**0.75**	**0.13**	***0.22***	***53***	***37***	***2.4***	***0.29***	***1.1***	***0.90***
0.8	Inferred	4	0.66	0.12	*0.18*	*65*	*39*	*2.5*	*0.22*	*1.0*	*0.93*
1.0	Measured	25	0.90	0.15	*0.22*	*54*	*39*	*2.5*	*0.22*	*1.0*	*1.07*
1.0	Indicated	12	0.74	0.16	*0.26*	*50*	*39*	*2.4*	*0.24*	*0.9*	*1.08*
1.0	**Meas + Ind**	**36**	**0.85**	**0.16**	***0.24***	***53***	***39***	***2.5***	***0.23***	***1.0***	***1.07***
1.0	Inferred	2	0.75	0.14	*0.20*	*57*	*42*	*2.4*	*0.13*	*0.8*	*1.04*

Notes: Any apparent arithmetic discrepancies are due to rounding
NiEQ = nickel equivalent, g/t = grams per tonne, ppm = parts per million Mt = million tonnes
**Sc grades estimated independently by OreWin Pty Ltd*

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[6] *NiEQ cut-off was calculated as NiEQ% = Ni% + (Co% X 2.95), based on assumed metal prices of US$4.00/lb Ni, US$12/lb Co, at USD:AUD exchange rate of 0.70. NiEQ was calculated on Ni and Co only, with no consideration for scandium or platinum.*

Note Relating to Scandium

Prior to 2014, scandium was regarded as one of several minor accessory elements present in the laterite profile at Sunrise. Since acquiring the Project, Clean TeQ has pursued its interest in the potential for development of the scandium mineralisation. The nickel and cobalt resource area contains a noteworthy endowment of background-grade scandium, as well as a minor contribution from localized zones of high-grade scandium that occur mostly, but not exclusively, around the periphery of the nickel/cobalt resource area. While the average scandium grade over the deposit is low, the potential to generate significant by-product revenues from scandium oxide production, for a relatively minor incremental cost, could add substantial by-product credits to a nickel/cobalt development.

In early 2016, Clean TeQ commissioned OreWin Pty Ltd to develop an independent resource estimate for high-grade scandium, with the aim of assessing whether the high-grade scandium may be considered as a standalone project. The results of this work were announced to the ASX on 17th March 2016. In mid-2016, Clean TeQ commissioned OreWin Pty Ltd to update the background scandium model throughout the nickel/cobalt resource area. The estimated scandium reported in the nickel/cobalt resource that is the subject of this current announcement is largely comprised of low-grade background scandium that is considered to be accessory to the nickel/cobalt resource and not standalone scandium resource in its own right. Only a small proportion (less than 10%) of the previously announced scandium Mineral Resource (17th March 2016) is contained within the current nickel/cobalt Mineral Resource at the quoted NiEq cut-off. The scandium Mineral Resource from 17th March 2016 remains unchanged.

There has been no historical production of nickel or cobalt from the Project area.

7 Geological Setting and Mineralisation

7.1 Regional Geology

The regional basement consists of sedimentary rocks of the Early to Middle Ordovician Girilambone Group (see Figure 7-1). The Australian Stratigraphic Units Database describes the Girilambone Group as deformed and metamorphosed, micaceous, quartzose and quartz-lithic sandstone, pelite, chert, minor conglomerate, siltstone, quartzite, mafic to intermediate volcanic rocks and black shale. The Girilambone Group was deformed during the Late Ordovician to Early Silurian period, during the Benambran Orogeny.

The Girilambone Group has been intruded by numerous Alaskan-type, mafic to ultramafic complexes, known as the Fifield Suite, which occur in a roughly north-south trending belt running from north of Condobolin to south of Bourke. These intrusions are typically zoned, with an ultramafic core surrounded, concentrically, by pyroxenite, monzonite, gabbro and monzodiorite.

The Project site is located approximately 5 km northwest of Fifield, at the eastern end of a mafic/ultramafic intrusion which is known as the Tout Intrusive Complex. The core of the intrusive body at deposit is dunitic, surrounded by pyroxenite and gabbro (see Figure 7-2).

Webb (2014[7]) states that the lack of internal fabric or significant alteration in the Tout Complex indicates that the intrusion at Fifield is related to a post-Benambran phase of extension-related magmatism. Webb carried out U-Pb SHRIMP dating of zircon from the Tout Complex gabbro, which indicated an Early Silurian age of 439.6Ma ±8.5Ma (mega-annum or million years).

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

The regional geology of the Fifield area is shown on NSW Geological Survey Narromine 1:250,000 geological map, Sheet SI/55-3 (Sherwin and Healy, 1997[8]).

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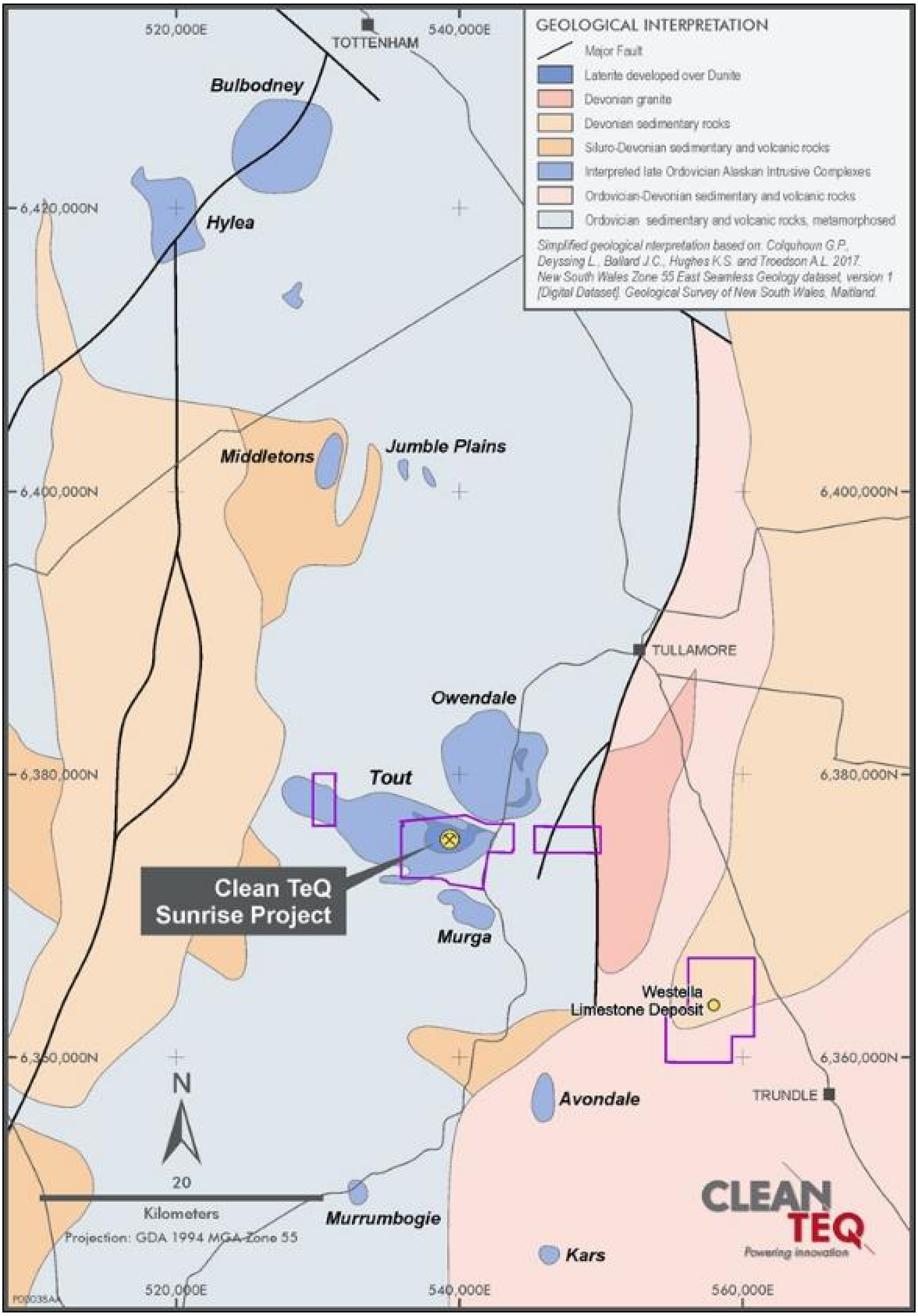
[7] Webb, K., 2014. The age and tectonometallogenic setting of the Fifield “Alaskan-type” intrusive suite, Lachlan Orogen, NSW. Bachelor of Science (Honours), School of Earth & Environmental Sciences, University of Wollongong, 2014.

[8] Sherwin, L., and Healy, M., 1997. Narromine 1:250,000 Geological Sheet SI 55-3 2nd Edition. NSW Department of Mines.



Simplified geological interpretation based on: Colquhoun G.P., Deyssing L., Ballard J.C., Hughes K.S. and Troedson A.L. 2017. New South Wales Zone 55 East Seamless Geology dataset, version 1 [Digital Dataset]. Geological Survey of New South Wales, Maitland.

Figure 7-1: District geological setting

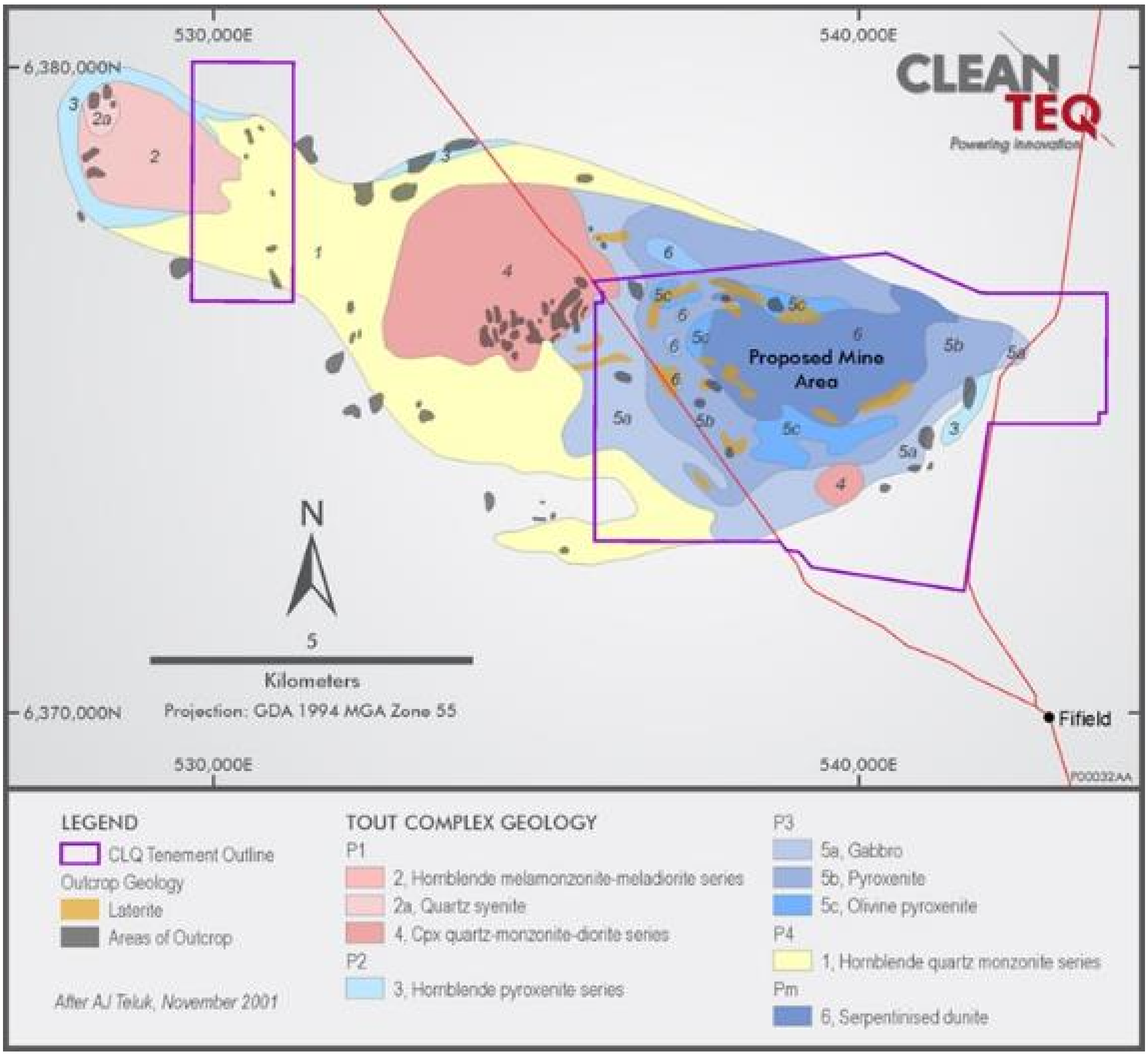


Figure 7-2: Intrusive Complexes in The Project Area

7.2 Local and Property geology

The Narromine 1:250,000 geological map shows a lack of outcrop near the Project, with small areas of laterite being the main feature of the proposed mining area. Most of this laterite is developed on small hills over the pyroxenite rim which surrounds the central dunitic core.

The deposit lies over a mafic to ultramafic intrusive complex known as the Tout Complex (see Figure 7-1 above), which is sometimes referred to as the Flemington Intrusion.

Figure 7-2 above illustrates the geology of the Tout Complex and shows the location of the Project tenure at the eastern end of the Tout Complex. In this location, the core of the intrusion is a serpentinised dunite, which is surrounded by pyroxenite and olivine pyroxenite, with gabbro forming an outer rim. Geological characteristics of the area are discussed in Elliot and Martin (1991[9]) and Teluk (2001[10]).

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[9] Elliott, S. J. and Martin, A. R., 1991. Geology and Mineralisation of the Fifield Platinum Province, New South Wales. Sixth International Platinum Symposium (Perth WA), Guidebook for the Pre-Symposium Field Excursion.

[10] Teluk, A., 2001. Fifield Platinum Project, NSW Australia and Platinum Metallogenesis, Ural – Alaskan Type Complexes. Technical Report for Rimfire Pacific Mining NL prepared by Geodyne Pty Limited [Unpublished].

The lateritic weathering profile in which nickel, cobalt and platinum accumulated, developed preferentially over the serpentinised dunite core of the intrusion, which is about 4 km east-west and 2 km north-south (see Figure 7-3). Figure 7-21 below (in following Section 7.2.5.1, Mineralisation) illustrates the distribution of the nickel, cobalt and platinum resource, high grade cobalt zones and high-grade scandium zones. Most of the high-grade scandium is associated with the surrounding pyroxenites, forming a rim around the nickel, cobalt and platinum resources.

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

The lateritic weathering profile has been divided into five distinct layers based on a combination of geology and geochemical data. These laterite layers, from top to bottom, are as follows:

- Residual Overburden (OVB)
- Transitional Zone (TZ)
- Goethite Zone (GZ)
- Silicified Goethite Zone (SGZ)
- Saprolite Zone (SAP).

The SAP overlies ultramafic bedrock. The bulk of the nickel and cobalt mineralisation occurs in the GZ and SGZ. The general nature of the profile is shown diagrammatically in Figure 7-3 and the zones are described in more detail below.

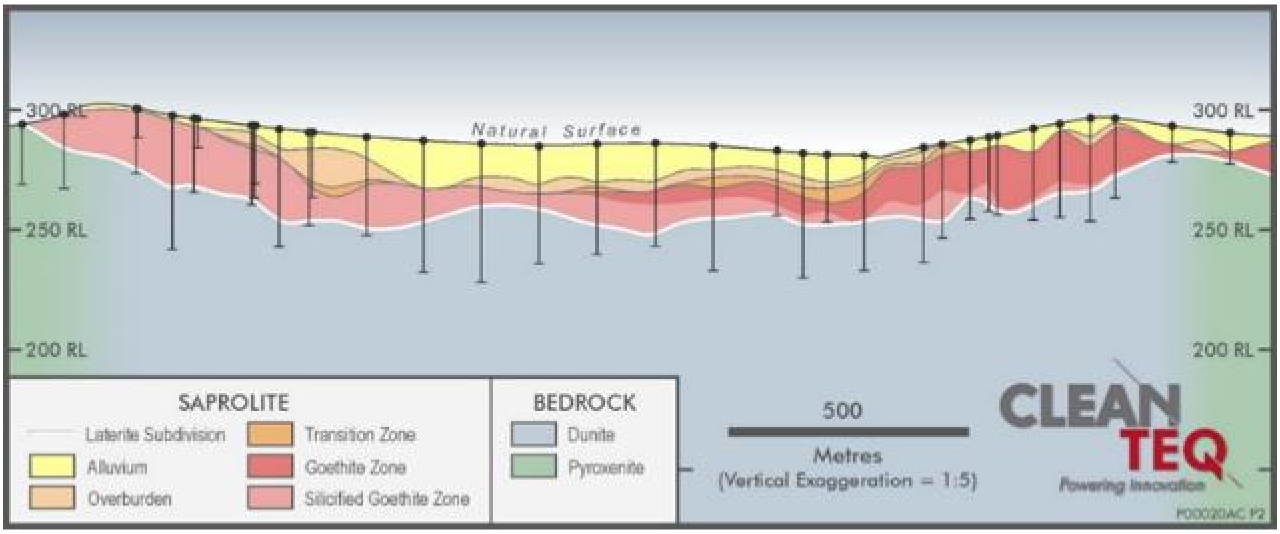

Figure 7-3: Schematic Geological Section

Several distinct Tertiary drainage channels cut across the deposit. These are typically 20 to 25 m deep but may be up to 40 m deep. These palaeo-channels are now filled with alluvium.

The lateritic profile is best developed on the old hilltops between the palaeo-channels. This pattern is typical of laterite development in present-day tropical environments. There may also have been erosion of the laterite along active drainage channels.

The main palaeo-channels are illustrated in Figure 5-1 above, showing east-west, northeast and southerly trends. The GZ tends to be very limited or absent under the main palaeo-channels, so that they effectively divide the deposit into several parts. The SGZ tends to persist across the whole area, but it is generally thicker below the palaeo-highs.

7.2.1 Alluvial cover

The thickness of the alluvial cover (AV) varies from 0->10 m. Where present, the alluvium consists mainly of sticky dark red haematitic clay (see Figure 7-4), with local gravel (see Figure 7-5) and minor lenses of ferruginous pisolite.

Mineral modelling (Brand, 2018[11]) has incorporated all available x-ray diffraction (XRD) data and shows that the main minerals present in the alluvial cover are kaolinite, goethite, haematite, smectite, carbonate, chlorite and white mica. Maghaemite pebbles have also been observed during logging.

The alluvial deposits generally do not contain significant nickel, cobalt or platinum, although similar alluvial deposits further south (near the township of Fifield) host some platinum 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 north-eastern parts of the area where magnesite mining has occurred in the past.

Figure 7-4: SDD014 0.5 m AV (Red Haematitic Clay)

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[11] Brand, N., 2018. Sunrise Mineralogy (and Chemical Validation). Powerpoint Presentation, 12 April 2018. Includes data from pXRF, NIR, VNIR, SWIR and Hylogger as well as FTIR and RAMAN (to detect MnOx) Metallurgy Department's XRD and XRF) (Draft).

Figure 7-5: SDD015 2.30 m AV (Mottled Goethite-Clay With mm to cm Sub-Angular Gravel and Pisolites)

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 above the pyroxenites where individual units are less clearly defined.

7.2.2.1 Overburden

The overburden zone (OVB) is commonly characterised by clay, gravel and ironstone. It varies from deep red haematitic gravel (which contains maghaemite), as illustrated in Figure 7-6 below, to light brown or green clayey intervals with scattered pebbles.

It is usually barren or very low-grade from an economic perspective. Short wavelength infrared (SWIR) scans, which were conducted in recent RC drill holes, indicate that montmorillonite is the most common clay mineral with occasional wedges of kaolinite.

Figure 7-6: SDD021 4.30 m OVB (Red-Brown Pisolitic Gravel and Ironstone – Haematite-Clay-Maghaemite)

7.2.2.2 Transition Zone

The transition zone (TZ) is strongly goethitic with minor to major kaolinite and gibbsite. Haematite and silica are present in minor amounts. SWIR has also indicated the present of low to medium amounts of smectite/nontronite, as well as minor chromite and magnetite. Pisolites are often observed, especially in the upper parts of this zone. The TZ grades downwards into orange-brown goethitic material. The upper part of the TZ is usually barren or very low-grade and will also be overburden from an economic perspective. Figure 7-7 and Figure 7-8 below provide a representation of the variability within TZ between and within holes. The TZ is also quite variable in thickness (up to about 15 m).

Figure 7-7: SDD020 5.37 m TZ (Medium-Hard, Brown to Red Goethite-Clay with Minor Haematite)

Figure 7-8: SDD014 7.5 m TZ (Brown to Orange Goethite-Clay with Minor Carbonate)

7.2.2.3 Goethite Zone

This goethite zone (GZ) is a relatively uniform, very fine grained, orange-brown, goethite rich layer. Texturally it resembles clay, but assays and XRD analyses have shown that it consists mainly of extremely fine goethite (see Figure 7-9). It typically contains more than 40% iron and low percentages of silicon, usually around 5-10%. Minor gibbsite and vermiculite have also been identified. Some sections of the GZ contain black manganese wad, which tends to smear the outside of the core (see Figure 7-10). High manganese is usually associated with elevated cobalt values. Other minor minerals which have been identified by XRD or SWIR in the GZ include haematite, chromite, smectite/nontronite, kaolinite and magnetite.

The upper boundary of the GZ appears to be quite gradational while the lower boundary is more abrupt, but potentially quite irregular. The thickness of the zone varies and is up to about 20 m.

Figure 7-9: SDD016 20.85 m 6.6.2.3 GZ (Brown to Yellow, Extremely Weathered Goethite with Minor Mica)

Figure 7-10: SDD017 22.20 m GZ (Brown to Black, Extremely Weathered, Goethite-Manganese-Clay)

7.2.2.4 Silicified Goethite Zone

The silicified goethite zone (SGZ) is distinguished from the overlying GZ by an abrupt increase in silica content, with silicone values usually more than 15% and averaging over 20%. Other minerals which have been identified in the SGZ are medium amounts of smectite/nontronite with trace amounts of manganese wad, haematite, chromite and magnetite. 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, which is similar to the overlying Goethite zone.

Past reports indicate that the SGZ is more consistently developed (or preserved) than the GZ, with a typical thickness of about 10-30 m. However, core drilled in 2017, above the serpentinised dunite core of the deposit, indicates that the SGZ can be extremely variable.

Where strongly silicified zones are present they vary in texture from hard vughy ‘honeycomb’ textured rocks (see Figure 7-11), to goethite with meshworks of fine quartz veins (see Figure 7-12), to hard silicified goethite and finally to soft goethitic material (see Figure 7-13). Field experience has shown that the silica meshworks can sometimes be coarsely vughy towards the base of the zone, causing some drilling difficulties.

Figure 7-11: SDD021 31.73 m SGZ (Yellow to Brown, Silica-Goethite-Magnetite, with Clay Between Silicified Portions)

Figure 7-12: SDD015 37.2 m SGZ (Hard, Brown to White, Siliceous Goethite, with Narrow Quartz Veins)

Figure 7-13: SDD014 30.75 m SGZ (Extremely Weathered Brown to Yellow Goethite-Clay, with White Patch of Quartz)

7.2.2.5 Saprolite Zone

The saprolite zone (SAP) represents the weathered top of the underlying ultramafic bedrock. It is usually a paler brown to grey colour and consists mainly of smectite/nontronite, magnesite, dolomite and serpentine. Small amounts of goethite and silica are also present (see Figure 7-14), along with traces of chromite, magnetite and haematite. Relict igneous textures are usually preserved, and patches of relatively fresh bedrock are often observed. Veins of magnesite and dolomite are very common (see Figure 7-15). The saprolite zone rarely contains significant nickel or cobalt values, but some elevated platinum grades occur on a very localised basis. The thickness of this zone is unclear because most drill holes stop after penetrating it for just a few metres.

Figure 7-14: SDD021 40.42 m SAP (Hard, Brown to Grey Serpentinitic Saprock – Goethite-Serpentine-Carbonate)

Figure 7-15: SDD016 43.60 m SAP (Goethite-Magnesite-Calcite, with Large Magnesite Vein and Narrow Calcite Veins)

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 quickly away 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 they moved onto the pyroxenite.

7.2.4 Bedrock

Fresh basement rocks have rarely been intersected in past drill holes, which generally terminated in the saprolite zone. However, the eight diamond core holes which were drilled in late 2017 aimed to drill to bedrock. Six of the eight holes ended in fresh, dark green-black serpentinite, one hole ended in weathered serpentinite and the eighth hole stopped short (in SGZ).

Fresh rock samples were collected from the six holes which reached bedrock and polished thin sections were made for petrological study. This work was carried out by Martin Gole and Associates and reported in Gole (2018[12]). The samples were all very similar, being partly to completely serpentinised dunites (see Figure 7-16). Gole states that "most of the olivine was altered to classic mesh-textured lizardite-brucite assemblages (see Figure 7-17). Magnetite was present with a range of trace to 6%, mostly occurring as scattered, very fine-grained inclusion within serpentine and some veinlets. Minute traces of probable awaruite were present, along with traces of sulphides in two sections. Chromite is scattered through all the rocks, mostly as euhedral grains (see Note: Width of Field = 2 mm, Figure 7-18).

Where sufficiently preserved, the original texture is adcumulate, with <5% space between former olivine grains or at triple point junctions. Based on the interpreted igneous textures and mineralogy the rocks were olivine-chromite adcumulates".

Relict igneous olivine was present in five of the samples, ranging from trace to 15% of the mineral assemblage. The holes with highest olivine percentages were from the western side of the deposit.

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[12] Gole, M. 2018. Petrography of Syerston Dunite Samples, Tout Complex, Fifield, NSW. Unpublished report prepared by Martin Gole, Geological Consultant, 8 Landor Road, Gooseberry Hill, Western Australia, Australia [Unpublished].

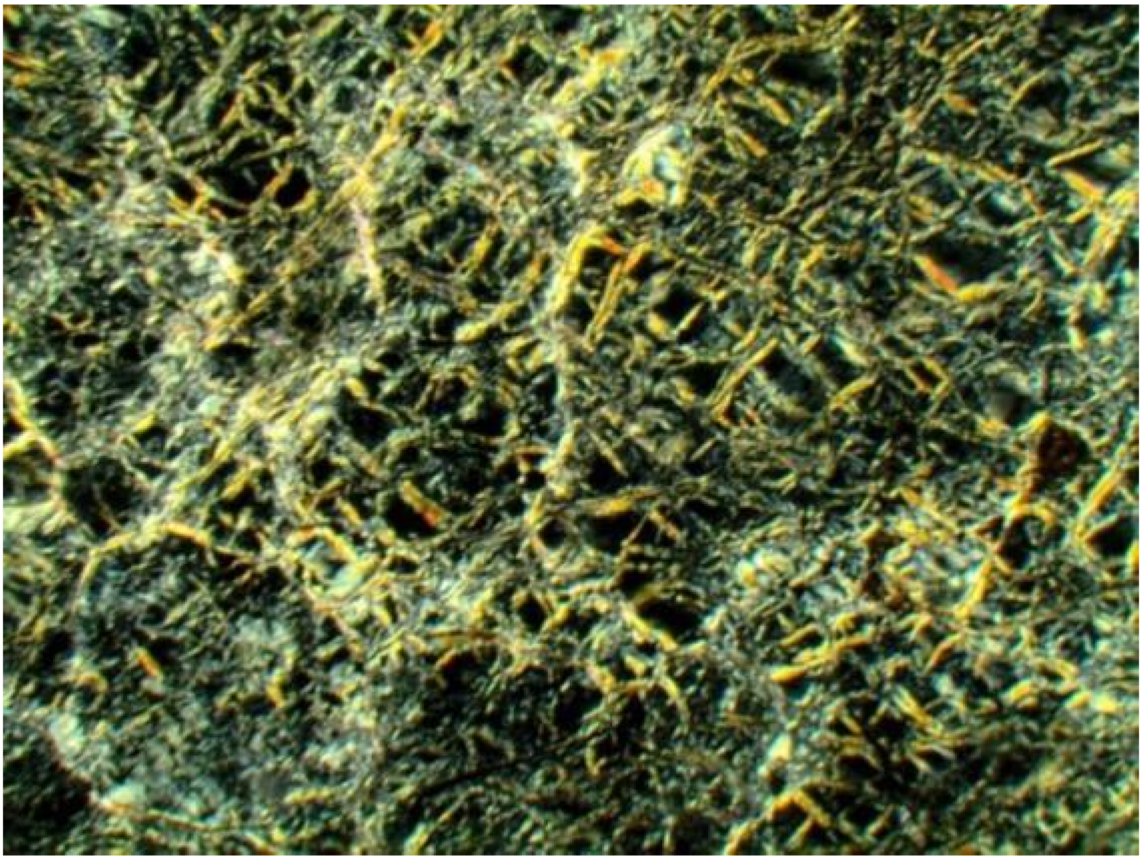


Figure 7-16: SDD014 - 67.66 m (Mesh-Textured Assemblage of Lizardite and Brucite with Minor Scattered Chromite. Former Olivine Grain Boundaries Marked by Pale Brucite-Rich Assemblages. Width of Field = 6.5 mm)

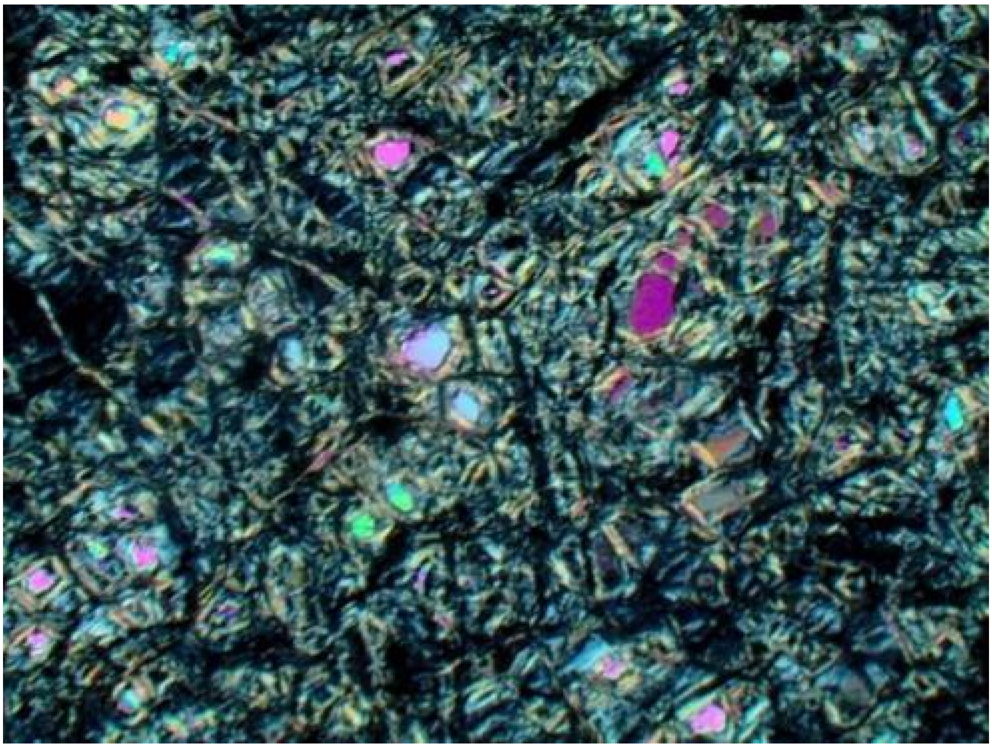
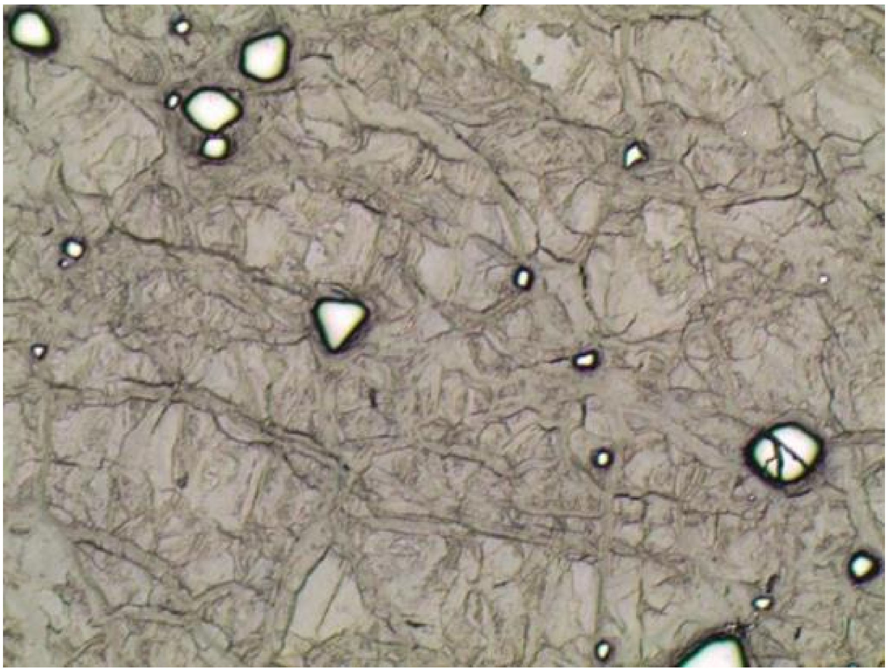


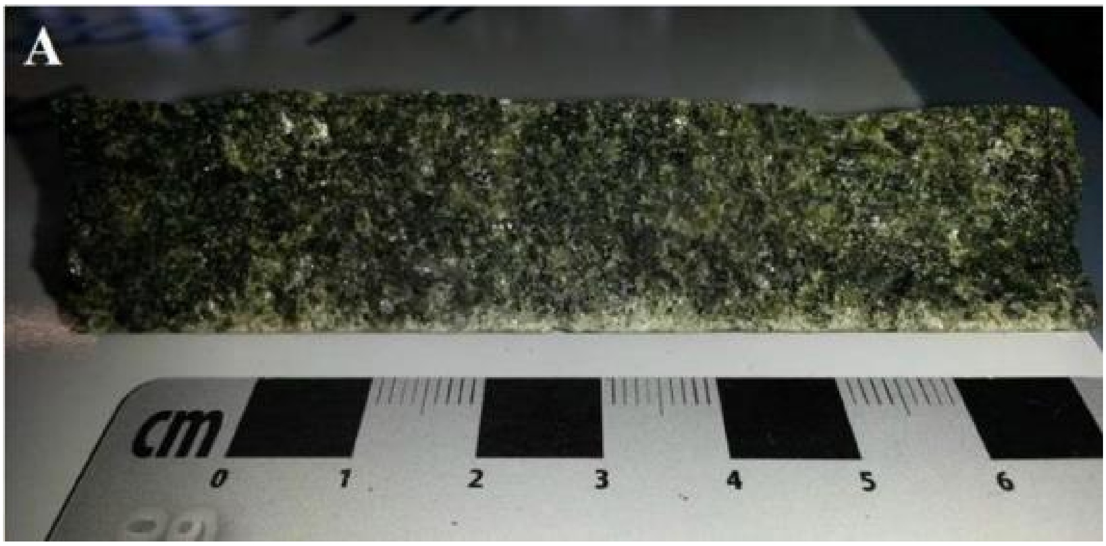
Figure 7-17: SDD021 - 52.35 m (Mesh-Textured Serpentine-Brucite Assemblage with Scattered Relict Olivine. Width of Field = 6.5 mm)



Note: Width of Field = 2 mm

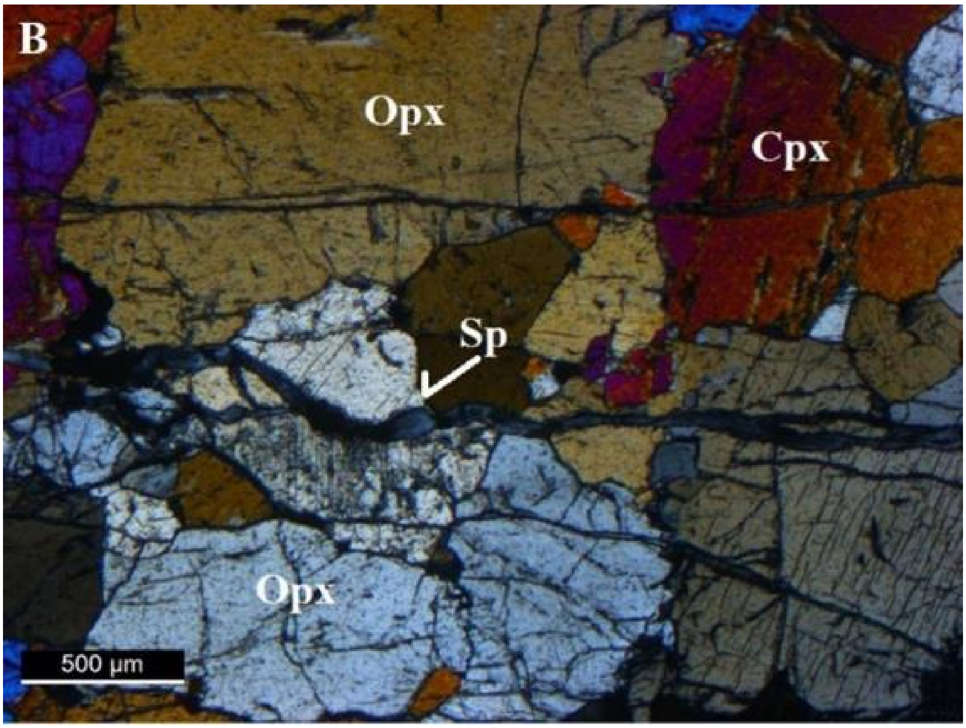
Figure 7-18: SDD017 - 66.52 m. (Reflected Light View of Mesh-Textured Serpentine-Brucite Showing Scattered Chromite Grains with Variable Grain Size)

Thin section petrology was also carried out by Webb (2014) on two samples from historical diamond hole FPD1 (identified by Webb as FFD1) which was drilled into pyroxenite in the north-western part of the Project areas. The two samples were collected from depths of 56 m and 63 m. In hand specimen, both were equigranular, medium grained and composed of a meshwork of green and black grains, with the black grains identified as pyroxene. The thin sections showed that the rocks were composed of approximately 70% clinopyroxene and orthopyroxene (in an 80:20 ratio) with approximately 10% altered olivine, 10% opaque minerals and 10% serpentine (as both veins and blebs). Clinopyroxene grains were usually larger than the orthopyroxene. Consisting of Subhedral to Anhedral Clinopyroxene and Orthopyroxene, With Minor Altered Olivine and Serpentinite Veining Figure 7-19 and Figure 7-20 below illustrate one of the FPD1 core samples, in hand specimen and thin section respectively.



Consisting of Subhedral to Anhedral Clinopyroxene and Orthopyroxene, With Minor Altered Olivine and Serpentinite Veining

Figure 7-19: Hand Specimen (A) of Pyroxenite From Drill Hole FFD1



Opx = Orthopyroxene, Cpx = Clinopyroxene, Sp = Serpentinite)

Figure 7-20: Photomicrograph (B) of Pyroxenite From Drill Hole FFD1 (With Subhedral to Anhedral Clinopyroxene and Orthopyroxene and Serpentinite Veining)

7.2.5 Mineralisation

7.2.5.1 Mineralisation – Sunrise Deposit

The Sunrise deposit is a nickel laterite deposit with higher than normal levels of associated cobalt and scandium, and local elevated platinum values. Potential ore grades are related to strong concentrations of iron oxides in the laterite profile, which has preferentially developed over the serpentinised 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 manganese-nickel silicate garnierite or clay minerals like smectite.

Figure 7-21 below depicts the Sunrise Deposit in GDA 1994 MGA Zone 55 coordinates, with the outline of the serpentinised dunite represented by the heavy blue line. This plan shows the distribution of the nickel-cobalt-platinum resource in pale blue shading, high grade cobalt areas in green and the distribution of high grade scandium resources is shown in red.

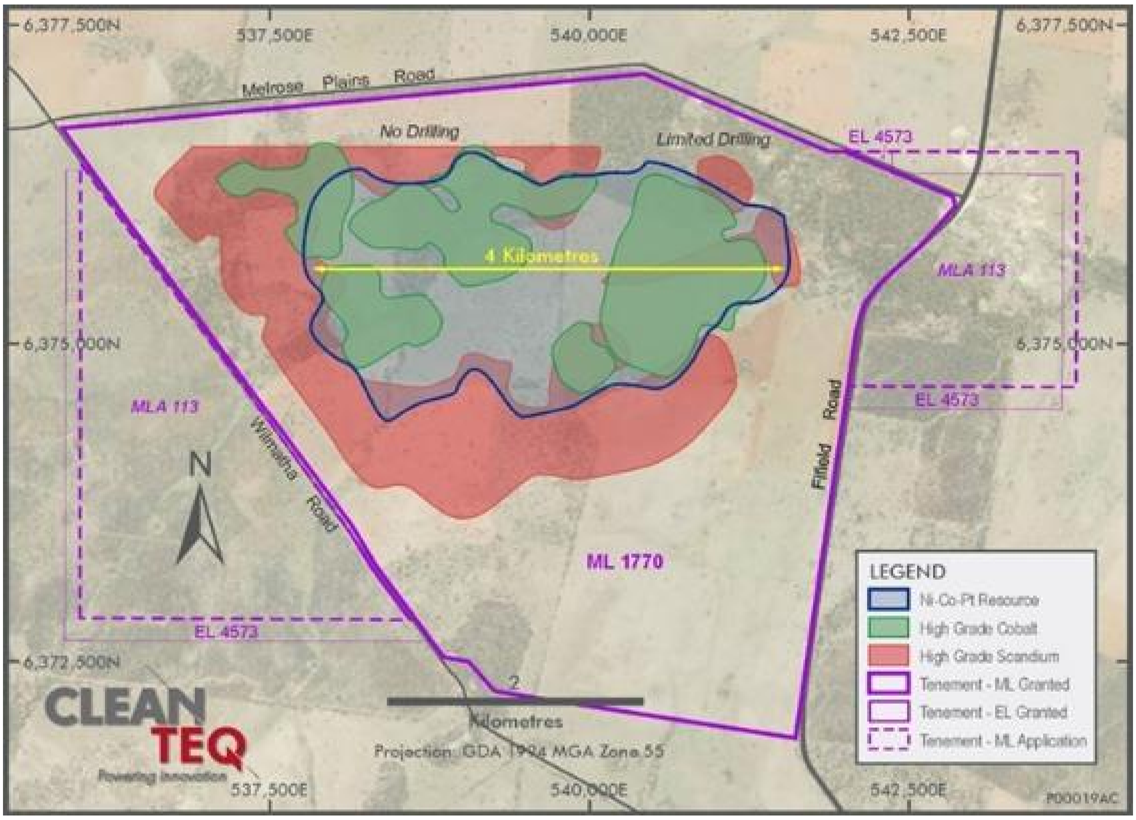


Figure 7-21: Plan Showing Distribution of High-Grade Resources

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

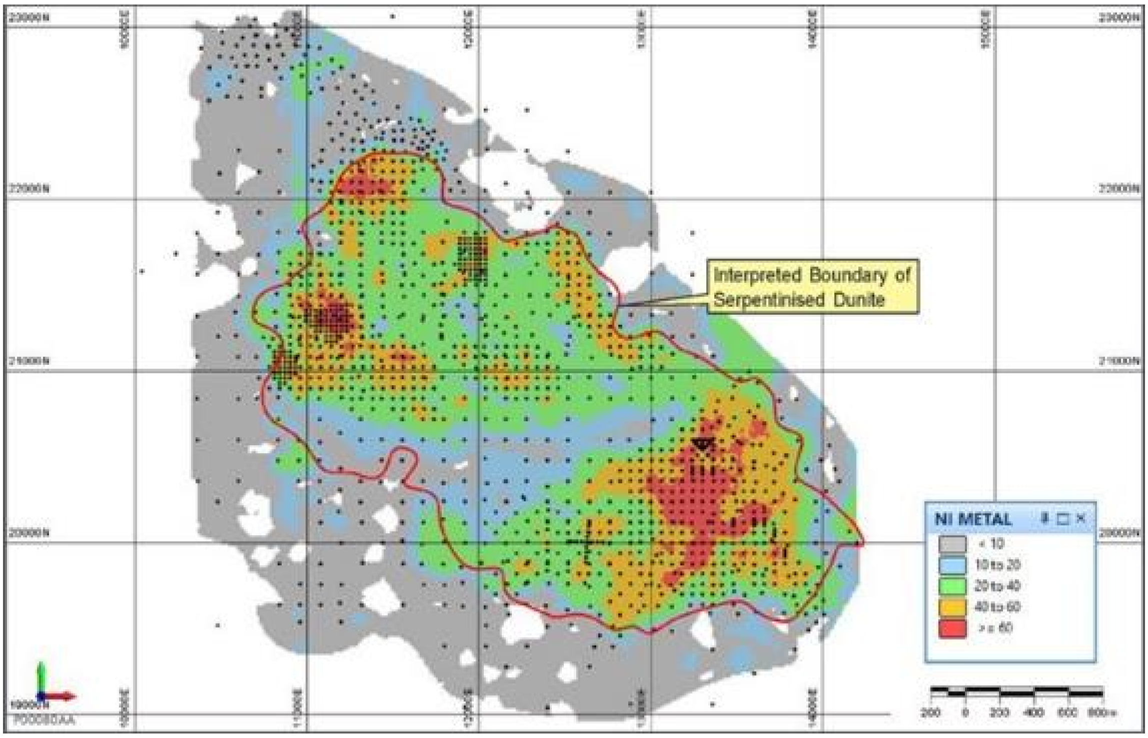
There are indications of some localised potential for bedrock platinum mineralisation in the underlying serpentinised dunite. However, there is no evidence of any nickel or cobalt sulphide mineralisation being present in the fresh rocks.

Nickel and cobalt mineralisation are intimately associated with iron oxides and, in the case of elevated cobalt values, also with minor manganese oxides, mainly in the following zones of the laterite profile:

- lower part of the TZ
- GZ
- SGZ.

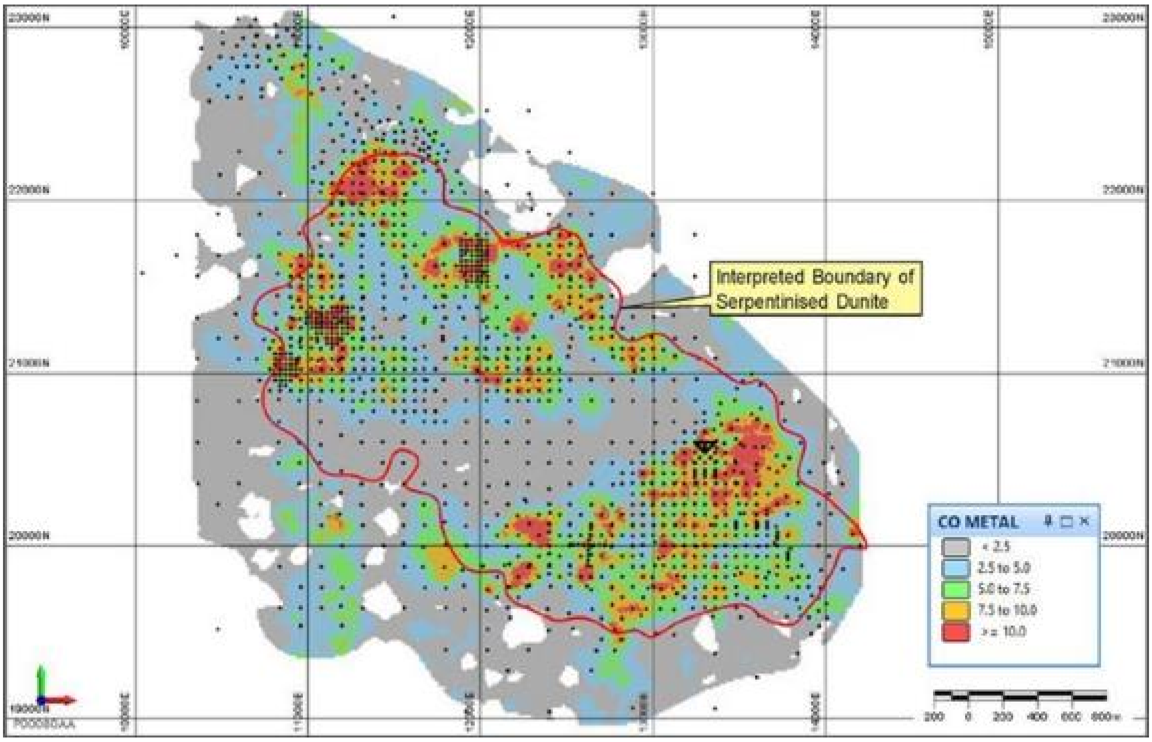
The highest nickel and cobalt grades occur in the GZ. Grades decline upwards from the base of the TZ, even though iron oxide content remains high. This is related to a change in iron oxide mineralogy from goethite to haematite. Grades are lower in the SGZ largely because of the substantial increase in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

Figure 7-22 and Figure 7-23 show plan views of the distribution of nickel and cobalt metal in the two main mineralised layers (GZ and SGZ) across the Sunrise Deposit. Note that these two Figures, as well as Figure 7-24 and Figure 7-25 are depicted in Local Grid.



Note: Legend shows Nickel Tonnes per 15mx15m Panel

Figure 7-22: Nickel Metal Distribution in GZ and SGZ



Note: Legend Shows Cobalt Tonnes per 15mx15m Panel

Figure 7-23: Cobalt Metal Distribution in GZ and SGZ

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

There appear to be differences in the spatial distribution patterns of high cobalt values in different parts of the deposit. In the southwest part, where extremely high values are less frequent, the cobalt tends to form relatively extensive, sheet-like layers within the GZ. In the north-western part, high cobalt values are more restricted in extent (but extremely high values are more common than in other areas) and they tend to occur as fringes around pronounced highs in the top of the SGZ. There are also indications in the north-western part of the deposit that some of very high cobalt concentrations might be localised in depressions in the upper part of the 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, the high cobalt occurs in numerous much smaller patches distributed over a larger area. This distribution pattern will need to be considered during the planning of pre-production grade control drilling, open pit optimisation and mine planning to avoid generating small starter pits that are heavily dependent on a single drill holes in areas of wide-spaced drilling.

In a broader sense, the lateral distribution of nickel and cobalt is heavily influenced by both the alluvium-filled Tertiary palaeo-channels which cut across the deposit and the extent of the underlying serpentinised dunite core to the intrusive complex.

The distribution of scandium is shown in Figure 7-24. Most scandium mineralisation occurs outside the interpreted boundary of the serpentinised dunite core, where elevated values mainly overlie pyroxenite.

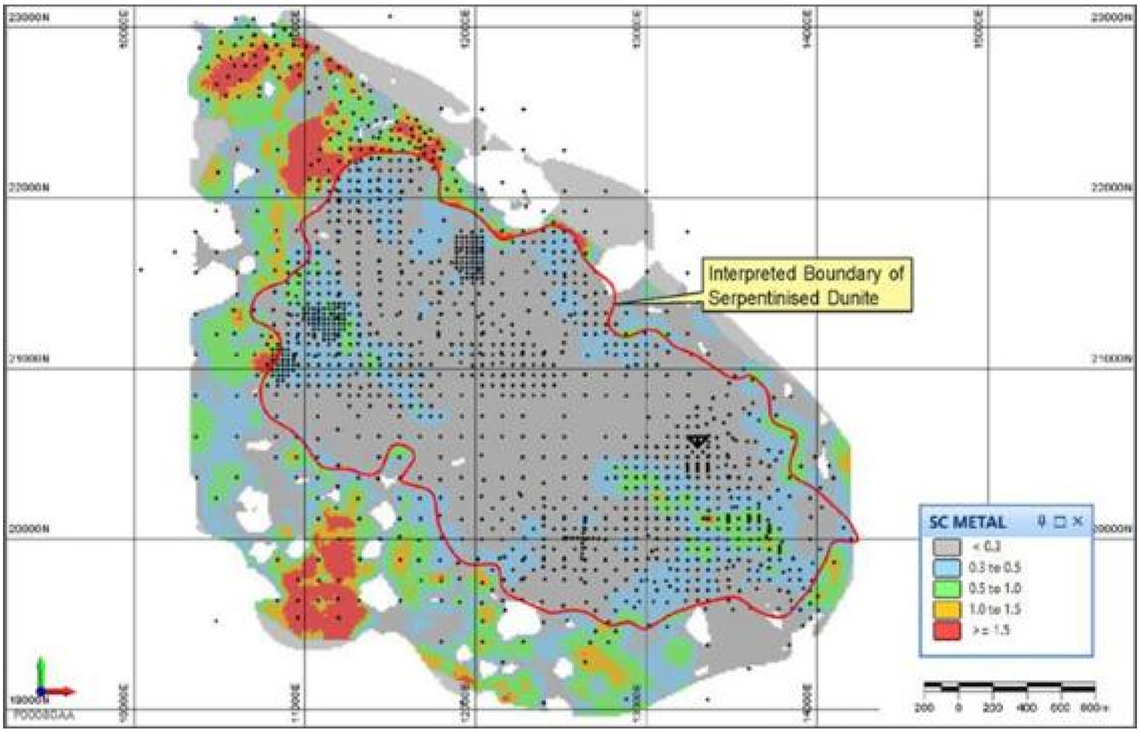
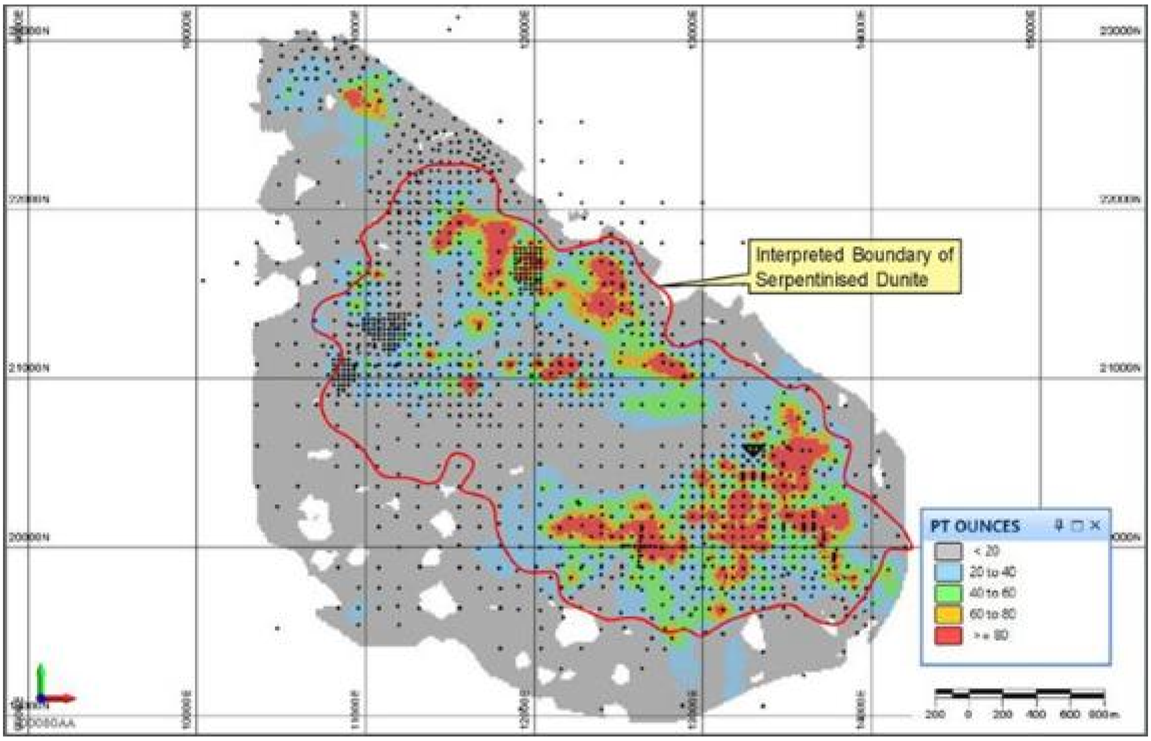


Figure 7-24: Scandium Metal Distribution in GZ and SGZ

Figure 7-25 shows the distribution of platinum in the Sunrise Deposit. Most elevated platinum values are located within the laterite profile above serpentinised dunite, and within this zone highest concentrations are seen in the east and the northwest. There is also one small area of elevated platinum values which is located further to the west and lies over pyroxenite.



Note: (Legend Shows Pt Ounces per 15mx15m Panel)

Figure 7-25: Platinum Metal Distribution in GZ and SGZ Mineralogy

Gangue

McDonald Speijers (2005a and 2005b) reported that the main mineral species identified in the various zones by previous XRD work are as summarised in Table 7-1 below (after SLA, 2000).

Table 7-1: Gross Mineralogy

Zone	Mineral	Relative Abundance
Transition Zone (TZ)	Goethite	Major
	Hematite	Minor
	Silica	Minor
	Gibbsite	Minor-Major
	Kaolinite	Minor-Major
Goethite Zone (GZ)	Goethite	Dominant
	Gibbsite	Minor
	Vermiculite	Minor-Major
Silicified Goethite Zone (SGZ)	Goethite	Dominant
	Silica	Dominant
Saprolite Zone (SAP)	Goethite	Minor-Major
	Silica	Major
	Serpentine	Major
	Smectite	Major

In recent mineralogical studies, Brand (2018) has generated a mineral model for Sunrise that is based on existing mineralogy reports, understanding of peridotite weathering, drill core observations, chemical analysis and SWIR analysis. This work is ongoing and has resulted in the identification of additional minerals. Figure 7-26 below above presents the current Mineralogical Model and shows the distribution and abundances of the minerals present in the overlying Alluvium and each of the LATZONES.

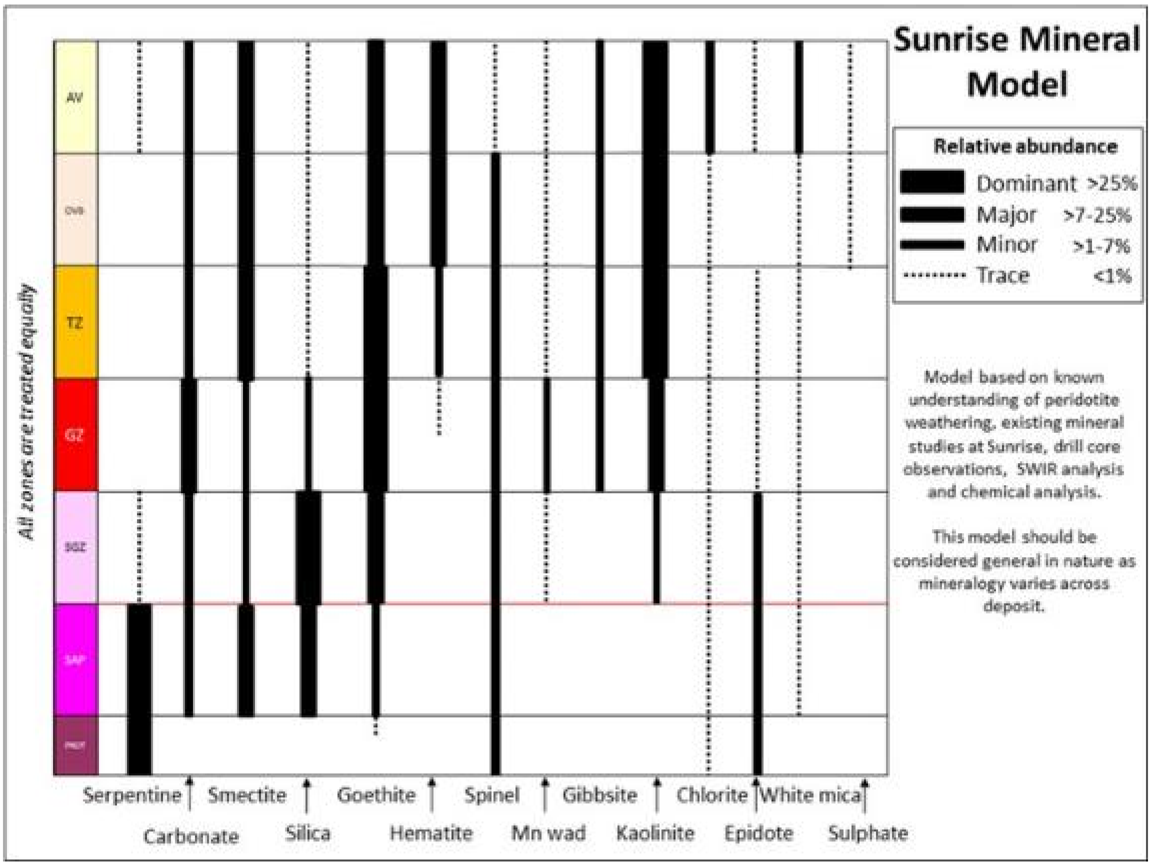


Figure 7-26: Mineralogical Model showing Mineral Distributions for The Main LATZONES

Nickel and Cobalt

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 nickel grades are in the goethite zone (which has the highest iron content, usually >40%), 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.

Elevated cobalt values tend to be closely associated with manganese oxides.

Platinum

Where platinum has been physically identified it occurs as grains of ferro platinum, but there is very little detailed mineralogical information for platinum in the resource area.

Examination of the spatial distribution of platinum grades on cross sections strongly suggests that:

- potential source areas in the bedrock are probably very localised and possibly steep dipping

- there has been substantial lateral dispersion of platinum values in the upper parts of the lateritic profile, particularly in the goethite zone.

This implies that extensive chemical remobilisation of platinum occurred during weathering, analogous to the secondary dispersion of gold in many Western Australian laterite deposits. This in turn raises the possibility that there might be mineralogical differences between residual and secondarily deposited platinum in different parts of the weathering profile, which could impact on metallurgical recovery of platinum.

Chromium

There is clear evidence from check assaying work that chromium becomes more readily F6-17-soluble going up through the weathering profile. This implies that the mineralogy of chromium also varies vertically.

Old mineralogical studies (e.g. Amdel, 1989[13]) reported that electron probe analyses showed that goethite contains significant chromium ranging up to 9.4% chromium(III) oxide (Cr_2O_3). It seems likely that in the upper parts of the weathering profile a significant proportion of the Cr occurs in secondary forms, intimately associated with iron oxides or perhaps to some extent with clay minerals. However, in the saprolite and bedrock chromium probably occurs as chromite grains.

Going up through the profile, it appears that a higher proportion of the chromium may not be amenable to extraction by gravity methods.

Scandium

Review of the scandium assay data across the deposit shows that it is pervasively distributed at elevated grades (>180ppm scandium) throughout the lateritic profile, but generally preferentially around the periphery of the central dunite complex (see Figure 7-24 above). The bulk of the scandium mineralisation is present within the OVB, TZ, and GZ. Mineralisation does persist into the SGZ and SAP, but generally at lower grades.

Zones of high-grade scandium mineralisation were interpreted using a notional cut-off grade of 180ppm scandium.

These high-grade scandium zones were broadly poddy over the entire project area, and they occurred across all horizons in the lateritic profile.

7.2.6 2017 Mineral Resource Estimate Revised Interpretation

A revised geological interpretation of the laterite zones recognised by McDonald-Speijers was carried out during 2017. 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.

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[13] Amdel – Mineralogical Analysis of Lateritic Ore from Fifield, NSW. Report to Poseidon Limited (?), in 6-Monthly Exploration Report for EL 2663 and EL 2664, Fifield Area – NSW, for the Period 23 September 1989 to 22 March 1990.

In addition to the specific chemical criteria outlined in Table 7-2 below, graphic columns for iron, silicon and aluminium were placed beside the hole trace in Figure 7-27 to assist in the interpretation by providing a visual comparison of the iron, silicon and aluminium cut-over values between the various zones in the laterite profile.

In Figure 7-27 below, the graphics (hatch) columns for iron, silicon and aluminium were depicted and distinct colour differences were very apparent between the zones. Within the GZ zone (dark orange hatch), the iron content (>40% iron: magenta) directly contrasts with the silicon content (<10%; yellow).

The laterite zones have distinctly different chemical signatures as depicted in Figure 7-27 below and discussed in the following text.

Figure 7-28 shows an extended version of cross section 11,500E with drill holes coloured to depict the interpreted LATZONES, with cobalt values plotted to the left of each drill hole trace and nickel values plotted to the right.

7.2.6.1 AV – Alluvium Zone

This zone remained as interpreted by McDonald Speijers and is characterised by high iron and silicon and very high aluminium. The nickel and cobalt values are extremely low with mean values of approximately 0.07% and 0.009% respectively.

7.2.6.2 OBV – Overburden Zone

The McDonald Speijers OVB zone was not revised. This zone is characterised by nickel values below 0.2%, very low cobalt values (<0.02%), silicon values like or slightly higher than the underlying TZ, but with relatively higher aluminium content. The OVB zone contains mean values for nickel and cobalt of 0.11% and 0.015% respectively.

Table 7-2: Chemical Signatures* Utilised - 2017 Resource Estimate Geological Interpretation

Element	Element Concentrations	Major Elements	Resource Elements	
Transition Zone – TZ				
Iron	Average 40%	Red	Nickel average 0.46%	Red
Silicon	Average 7%	Orange	Cobalt average 0.03%	Grey
Aluminium	Average 5%	Magenta	-	-
Manganese	Average 0.2%	-	-	-
Chromium	Average 4,800ppm	-	-	-
Goethite Zone – GZ				
Iron	>40%	Magenta	Nickel average 0.75%	Magenta
Silicon	5-10%	Orange	Cobalt average 0.17%	Magenta
Aluminium	Average 4%	Red	-	-
Manganese	Average 1.2%	-		-
Chromium	Average 4,100ppm	-	-	-
Silicified Goethite Zone – SGZ				
Iron	Average 24%	Red	Nickel average 0.58%	Red
Silicon	>20%	Magenta	Cobalt average 0.07%	Red
Aluminium	Average 1.3%	Orange	-	-
Manganese	Average 0.5%	-	-	-
Chromium	Average 4,400ppm	-	-	-
Saprolite Zone – SAP				
Iron	<10%	Grey	Nickel average 0.28%	Orange
Silicon	Average 25%	Magenta	Cobalt average 0.02%	Grey
Aluminium	<1.0%	Grey	-	-
Manganese	Average 0.17%	-	-	-
Chromium	Average 3,000ppm	-	-	-

*Note: The chemical signature matrix was used as a guide only for the geological interpretation.

7.2.6.3 TZ – Transition Zone

The TZ represents weathered GZ material and was defined by aluminium values as they increase significantly within the TZ from 2-3% aluminium to >4%. The nickel values are typically below 0.46% nickel and cobalt values are below 0.03% (compared with the nickel and cobalt values in the underlying GZ, which are 0.75% and 0.17% respectively). The mean values of the TZ for nickel and cobalt are 0.36% and 0.04% respectively.

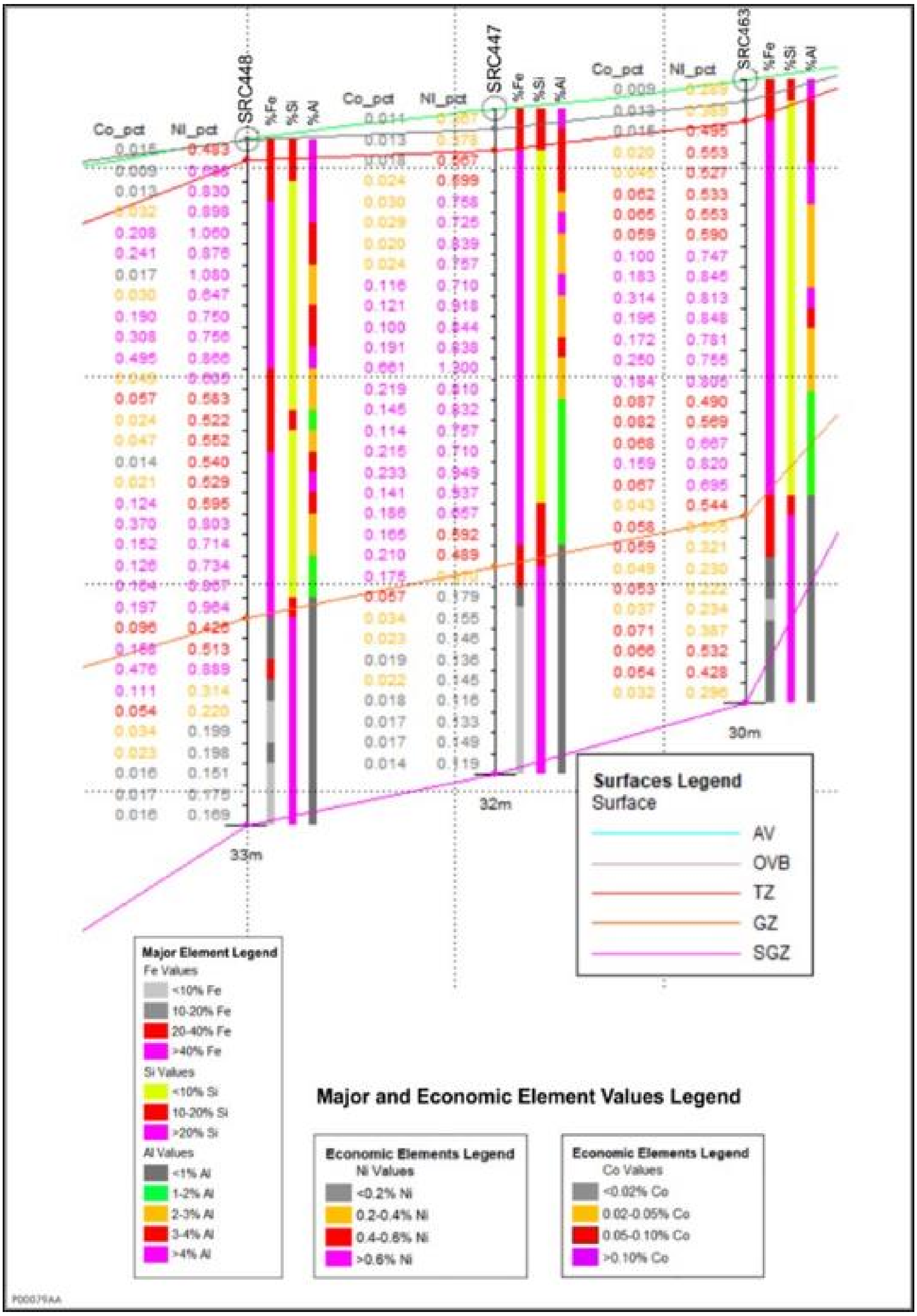

Figure 7-27: Portion of Section 10,500E Showing LATZONES, Ni, Co, Fe, Si and Al Percentages

These criteria resulted in the TZ/GZ boundary being lifted relative to that determined by McDonald Speijers, who used manganese values >0.35% and iron values >33% as the principal criteria for determining the TZ/GZ boundary.

7.2.6.4 GZ – Goethite Zone

The GZ is characterised by high iron and low silicon, with variable aluminium values. The most significant difference is the increase in nickel and cobalt values, where the mean nickel and cobalt values are 0.75% and 0.15% respectively.

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

7.2.6.5 SGZ – Silicified Goethite Zone

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

7.2.6.6 SAP – Saprolite Zone

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

Nickel and cobalt are lower than in the overlying SGZ, with the mean nickel and cobalt values being 0.25% and 0.025% respectively.

7.2.6.7 Comments

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

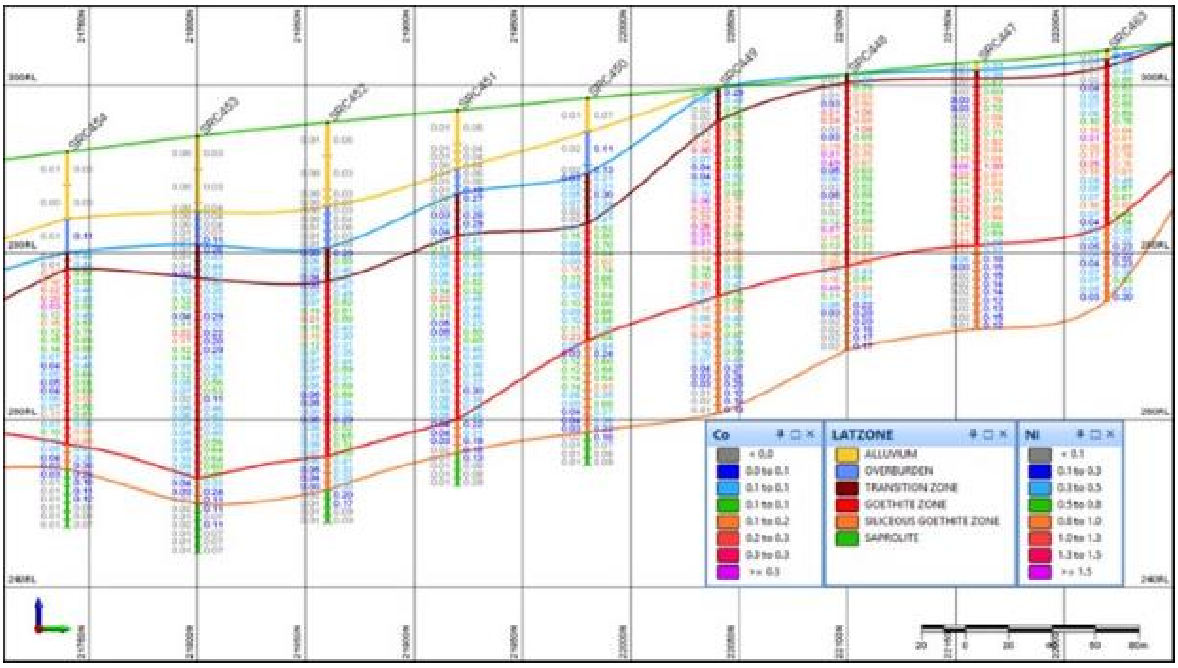

Figure 7-28: Section 11,500E With Geological Interpretation and Implicit Modelling Surfaces

The revised interpretation recorded these localised rises in the SGZ contact, but for providing a mining boundary 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 surface modelling (IM) process. Figure 7-28 above depicts the outcome of the IM procedure, which honours the data extremely well. The IM surfaces were used to constrain the resource block model.

7.2.7 Geological Modelling

The geological modelling process uses the geological interpretation as the framework for the creation of the geological surfaces. The subdivisions of the regolith profile have historically been referred to as zones and this terminology has been maintained during the current geological interpretation and modelling procedures.

The historically-identified zones which were geologically modelled are as shown in Table 7-3.

Table 7-3: Historically Identified Zones

Zone Abbreviation	Zone
Surface	Nominated Zone
AV	Alluvial Zone
OVB	Overburden Zone
TZ	Transition Zone
GZ	Goethite Zone
SGZ	Silicified Goethite Zone
SAP	Saprolite Zone

The geological interpretation strings on the north-south cross sections were filtered by their zone codes and then initially linked using ‘tie lines’, to ensure that the triangulation process which is used to form DTMs could proceeded without the individual triangles overlapping.

The ‘tie lines’ forced the triangulation process in the DTM creation to follow a specific linking sequence. Various triangulation processes including maximum volume, equiangular, maximum surface area and proportional triangles were tried in order to achieve a triangulated surface with no intersecting triangles.

A ‘Validate DTM’ routine was used to check for self-intersections, closures, invalid topology and surface integrity. If any of these issues were identified, the Wireframe Clean process was used to clean and repair the wireframe (DTM). If this process did not achieve the desired outcome, then the individual conflicting triangles were removed and replacement triangles were inserted.

To test the integrity of the DTM surfaces, the Assign function was used on an interval file (for example the assay file) and if during the assignment process a malformed triangulated surface was encountered then a warning sign was displayed. Through an iterative process, all intersecting or malformed triangles were removed and replaced by correctly formed triangles until no error messages were recorded.

The DTM surfaces of the geological model were made available to the Resource Geologist to use in the resource modelling process where the entire region between the zones was filled with blocks to create a rock model representing the material occupying the volume between the surfaces.

8 Deposit Types

The deposit 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. A typical laterite profile is shown in Figure 8-1, from Brand (2018. This figure also depicts Clean TeQ's interpreted geological zones, Alluvium, Overburden and Laterite zones (GZ, SGZ, SAP), as well as a typical drill hole profile showing MgO% with depth.

The bulk rock geochemistry, mineral chemistry and internal zonation of rock types is consistent with Alaskan-type mafic-ultramafic intrusive complexes.

Nickel and cobalt mineralisation is intimately associated with Fe oxides, and in the case of elevated Co values also with Mn oxides, mainly in the following zones of the laterite profile:

- Lower part of the TZ
- GZ
- SGZ.

There is no evidence of any Ni or Co sulphide mineralisation being present in fresh bedrock.

The highest nickel and 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, infill drilling has confirmed that unusually high Co grades (above about 0.4% - 0.5% Co) tend to be localised, with poor lateral continuity. 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.

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 north-western 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. There are also indications in the north-western part of the deposit that some of very high Co concentrations might be localised in depressions in the upper part of the 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.

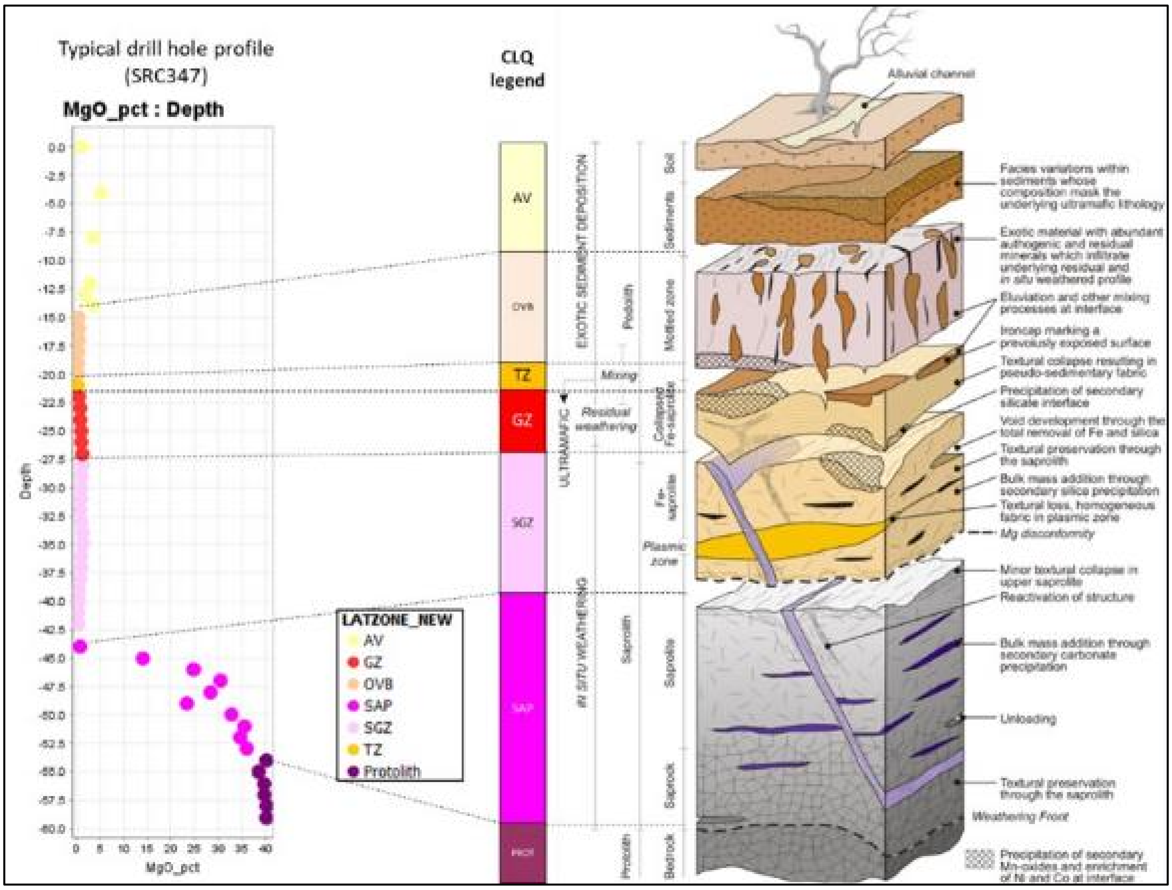

Figure 8-1: Regolith profile

In a broader sense, the lateral distribution of nickel and cobalt is heavily influenced by both the alluvium-filled Tertiary palaeo-channels that cut across the deposit (Figure 5-1) and the extent of the underlying dunite core to the intrusive complex.

9 Exploration

9.1 Grids

A local project grid was apparently established in 1986. McDonald Speijers (2005a and 2005b) stated that they had not sighted any information regarding the survey procedures or controls that were originally used. Old plans (c.1988) indicate that the origin point of the grid was in the southwest corner of the area and was initially assigned coordinates of 0mE and 0mN. This point apparently equates to current grid coordinates of 10,000mE and 20,000mN.

Between 1990 and 1993 a constant of 10,000 m was added to local grid easting and northing coordinates. This modification appears to have remained in place until late 1998, when a further 10,000 m was added to northings only.

These changes in grid coordinates would have increased the risk of some drill hole coordinates being incorrectly reported, although McDonald Speijers were not aware of any cases where this appears likely to have happened.

Local survey control was apparently re-established for Black Range Minerals between November 1998 and February 1999 by licensed surveyors Terra Sciences Pty Ltd (now Geolyse Pty Ltd), based in Dubbo. Their report (Terra Sciences Pty Ltd, 1999[14]) indicates that four concreted local control stations had previously been set up in the area by Langford and Rowe of Dubbo, who had established Australian map grid (AMG) coordinates for each control station. McDonald Speijers stated that they had not seen any details about when or how these stations were originally marked out, although Terra Sciences reported in March 1999 that Langford and Rowe had used two permanent state survey marks at Tullamore to coordinate the local stations by GPS. Terra Sciences stated that they adopted the AMG coordinates of the four concrete blocks previously determined by Langford and Rowe.

A local control traverse by Terra Sciences showed lateral and vertical errors at the local stations to be less than 0.01 m.

In their March 1999 report, Terra Sciences noted that "...large and inconsistent errors existed in the Local Coordinate system and the source of these errors could not be located despite exhaustive research". However, no details were provided about the nature or magnitude of the errors. Terra Sciences re-established the local coordinate system. The exact relationships between old and new Local Coordinates were apparently unclear.

A transformation between local grid and AMG coordinates was established by Terra Sciences in 1999, "as much as available information allowed" (see Table 9-1). The AMG coordinate system used was not specified in any reports which McDonald Speijers had sighted, but their direct enquiries with Terra Sciences (now Geolyse Pty Ltd) confirmed that it was AGD84.

McDonald Speijers stated that “the Project needs to remain aware that a new Australian coordinate system known as the Geocentric Datum of Australia (GDA94) has now been adopted”. All official survey data (e.g. mineral tenement coordinates) are now reported in the new system, which is offset from AGD84 coordinates by approximately 200 m to the northeast. This creates potential for errors and care will be required to ensure that any AMG coordinates used in connection with the project are in the same system.

All coordinates used in McDonald Speijers report referred to the Local Grid.

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[14] Terra Sciences Pty Ltd., 1999. Survey Report, Syerston Project 1998-1999. Internal Report to Black Range Minerals, March 1999 [Unpublished].

Table 9-1: Local Grid to AGD84 Transformation

Point	Local Grid		AGD84	
	East	North	East	North
A	12,023.806	19,893.756	539,656.544	6,374,455.712
B	13,018.194	21,215.244	539,707.433	6,376,108.043
Distance	1,653.825	1,653.114	-	-
Bearing	36º57'38"	1º45'51"	-	-
Rotation	-35º11'47			
Scale Factor	0.99957			

It appears that since the beginning of 1999, survey accuracy has probably been good on a local basis, although the level of accuracy of the local to AGD84 grid transform is unclear.

Surveys for holes drilled prior to about November 1998 are not adequately supported by surviving documentation. Consequently, McDonald Speijers could not be confident that they would have satisfied normal accuracy requirements for resource estimation purposes (typically about ±0.1-0.2 m).

9.1.1 Drill Hole Surveys (Up To 2015)

Collar survey procedures prior to late 1998 are not clear and McDonald Speijers stated that they had not been able to locate any survey reports for that period. This does not create a significant technical risk, since most of drill holes used for resource estimation were drilled after 1988. However, some limited areas of the deposit are still dependent on pre-1999 drill holes.

Collars of holes drilled between the end of 1998 and the end of 2000 were marked out in advance and picked up again after completion by licensed surveyors using Total Station instruments, referencing to the concreted local control marks. Collar positions were generally reported in both local and AGD84 coordinates.

In 2005, most drill hole collar locations were marked out accurately in advance by licensed surveyors. Holes were collared as close to the pegs as possible, generally within 0.1 m. If it was necessary for a hole to be offset from the pegged position, the local grid offset distances were measured to the nearest 0.1 m by the rig geologist, using a steel tape and the coordinates adjusted accordingly. McDonald Speijers were satisfied that this was sufficiently accurate for resource estimation purposes.

Some twin holes and close-spaced holes near previous high-grade platinum intercepts that were drilled late in the 2005 program were picked up by the surveyors after completion.

Documented drill hole collar surveys are summarised in Table 9-2.

Table 9-2: Documented Drill Hole Collar Surveys

Surveyors	Survey Date	Range of Hole Numbers
Terra Sciences	November 1998 – February 1999	SAC143,145,147 SDD6 No other recognisable hole numbers
Terra Sciences	June – October 1999	SRC418-1004
Terra Sciences	December 1999	SCW7-9 SRC1005-1060
Geolyse Pty Ltd (formerly Terra Sciences)	January 2005	SRC1077-1193
Geolyse Pty Ltd	April 2005	SRC1194-1249

In some cases, coordinates in these reports were only listed against pre-drilling site reference numbers that do not match drill hole numbers in the database, but many of these reference numbers are recorded together with the hole numbers on the original logs or on other surviving work sheets.

For holes drilled before November 1998, the only ‘source’ documents which can now be used to validate collar locations are the original drill hole logs, which only show approximate local grid coordinates. McDonald Speijers noted differences of up to 40 m or more between these and the local coordinates in the database.

No down hole surveying appears to have been carried out. The database shows all holes to have been vertical, with an average depth of about 35 m. It was reasonable to assume that they remained vertical and the lack of down hole surveys is consistent with normal industry practice in these circumstances.

9.2 2017 Survey Validation

In the establishment of the new Clean TeQ Geobank™ database (CLQGB_MODEL), the collar positions for the GB_SITE_SURVEY table were initially derived from the Black Range Minerals SYERSTON_DB Access database. There was some uncertainty as to the accuracy of the collar locations and every effort was focussed on identifying any records providing original survey data of the collar positions.

Licenced Surveyors Geolyse Pty Ltd, were approach to provide all records they retained of the surveying of the drill hole collars. They provided several Microsoft Excel™ workbooks, which were related to the setting out of proposed drill holes but not the actual drill hole identification. Within a few of these workbooks there were recorded pickups of the actual drill hole locations (see Table 9-3).

These Microsoft Excel™ workbooks were imported into Micromine™. Several Adobe Acrobat™ PDF files which had been converted to rich text were also imported in Micromine™. The coordinates in Local, AMG84 Zone 55 and MGA Zone 55 were input into the Micromine™ collar coordinate compilation and for each coordinate system it was established whether they had been either surveyed or calculated.

Most of the holes were surveyed by Differential GPS (DGPS) either as AMG84 Zone 55 or MGA Zone 55 and all the drill holes were converted to Local coordinates using a transformation function using a single point, a rotation and a scale factor.

Within Micromine™, the surveyed coordinates in both AMG84 Zone 55 and MGA Zone 55 were compared with the calculated local coordinates by creating a difference field (ΔE, ΔN and ΔRL) and reviewing the difference to ascertain whether there were any substantial differences that required either reviewing from the original source or re-surveying (if possible).

The original AMG84 to local transformation used two points, but this was rejected in favour of a single point lying between Geolyse survey control stations 9401 and 9404 and using a rotation and scale factor in the conversion. The single point used for the transformation cannot physically be located at this juncture.

The MGA to local transformation used the Geolyse survey control station 9404 together with a rotation and a scale factor. The transformation has been used to date to convert the drill hole collars picked up in MGA zone 55 to local coordinates. A recent ground inspection identified the approximate location of the survey control point 9404 but it had been destroyed by ploughing.

The AAM Group endeavoured to locate survey control point 9404 when they carried out the ground control for the recent light detection and ranging (LiDAR) survey but failed to locate the point despite being requested to include the station in the survey control network.

Table 9-3: Summary of Collar Survey Details

Date	Operator	Hole Type	Hole Number Range	Number of Holes	Local	Survey/ Calculated	Australian Mapping Grid, 1984 (AMG84)	Survey/ Calculated	MGA	Survey/ Calculated
Mar 1988 – Mar 1989	Noble – Poseidon JV	RAB	FR1–106	105	-	-	-	-	-	-
		RAB	FRRS1-82	82	-	-	-	-	-	-
		RAB	P30 & P36	2	-	-	-	-	-	-
Sep 1989 – Mar 1990	Noble – Poseidon JV	RAB	FR107-115	10	-	-	-	-	-	-
Sep 1991 – Sep 1992	Noble – Poseidon JV	DDH	FPD1-2	2	-	-	-	-	-	-
Aug 1993 – Aug 1994	Noble Resources	RAB	SRB117-119	3	-	-	-	-	-	-
		AC	SAC120-139	20	Yes	Survey	Yes	Survey	Yes	Calculated
Aug 1995 – Aug 1996	Uranium Australia	AC	SAC140–267	128	Yes	Survey	Yes	Survey	Yes	Calculated
Aug 1997 – Aug 1998	Uranium Australia	RC	SRC001–340	341	Yes	Survey	Yes	Survey	Yes	Calculated
		DDH	SDD1-5	5	Yes	Survey	Yes	Survey	Yes	Calculated
Aug 1998 – Oct 2000	Black Range Minerals	RC	SRC341-1004	732	Yes	Survey	Yes	Survey	Yes	Calculated
		RC	SRC1005–1060	56	Yes	Survey	Yes	Survey	Yes	Survey
		RC	SRC1065-1076	12	Yes	Survey	Yes	Survey	Yes	Survey
		DDH	SDD6–13	8	Yes	Survey	Yes	Survey	Yes	Calculated
		Calweld	SCW1–6	6	Yes	Survey	Yes	Calculated	Yes	Calculated
		Calweld	SCW7–9	3	Yes	Survey	Yes	Survey	Yes	Survey
Feb 2005 – Mar 2005	Ivanplats-Syerston	RC	SRC1077–1193	117	Yes	Survey	Yes	Survey	Yes	Calculated
		RC	SRC1194-1225	32	Yes	Survey	Yes	Survey	Yes	Calculated
		RC	SRC1226-1262	37	Yes	Survey	Yes	Survey	Yes	Calculated
Aug 2014		RC	SRC1263-1276	14	Yes	Calculated	Yes	Survey	Yes	Survey
Apr 2015	Scandium21	RC	SRC1277-1310	34	Yes	Calculated	Yes	Survey	Yes	Survey
Nov 2015		RC	SRC1311-1368	58	Yes	Calculated	Yes	Calculated	Yes	Survey
Feb 2016		RC	SRC1369-1383	15	Yes	Calculated	Yes	Calculated	Yes	Survey
Jul 2017		RC	SRC1384-1417	34	Yes	Calculated	Yes	Calculated	Yes	Survey
Sep 2017		RC	SRC1418-1427	10	Yes	Calculated	Yes	Calculated	Yes	Survey
Oct-Nov 2017		DDH	SDD014-021	8	Yes	Calculated	Yes	Calculated	Yes	Survey
Jan-Feb 2018		RC	SRC1428-1552	125	Yes	Calculated	Yes	Calculated	Yes	Survey

9.3 Topography

A photogrammetric topographic survey was apparently conducted by Geo-Spectrum (Australia) Pty Limited in 1999, using photography flown in September 1999 at a scale of 1: 20,000. McDonald Speijers were provided with a wireframe model of the topography which they presumed to have been generated from this photogrammetric survey. McDonald Speijers checked drill hole collar elevations in the database against the topographic wireframe during the review phase in early 2005 and found that:

- differences in elevation ranged from -2.64 to +5.58 m, but the average difference was only +0.17 m (see Table 9-4).
- differences increased with the age of the holes (see Figure 9-1).

In simple terms, RAB holes were the earliest phase (average difference +0.99 m), followed by Aircore (average +0.17 m), with RC being the most recent (average difference +0.02 m). This appears to reflect improved levels of survey control for drill holes since 1999. RAB holes and most AC holes were excluded from use in resource estimation

for the more recent RC holes and the majority of the Aircore holes, collar elevations were generally within about ±0.5 m of the topographic wireframe (see Figure 9-2).

McDonald Speijers stated that, in their experience, this is within a normal range of difference for these types of survey data. McDonald Speijers noted that most of the larger differences for recent holes were in areas which had been disturbed by old magnesite mining operations.

Table 9-4: Differences Between Drill Hole Collar Elevation and Topographic Wireframe Summary Statistics (January 2005)

Dataset	Number	Minimum	Maximum	Mean
All holes on wireframe	1446	-2.64	5.58	0.17
RC holes	1069	-1.42	1.33	0.02
Aircore holes	148	-1.74	4.22	0.17
RAB holes	207	-2.64	5.58	0.99
Used in 2000 Resource	1054	-1.42	4.22	0.04
Not used in 2000 Resource	392	-2.64	5.58	0.52

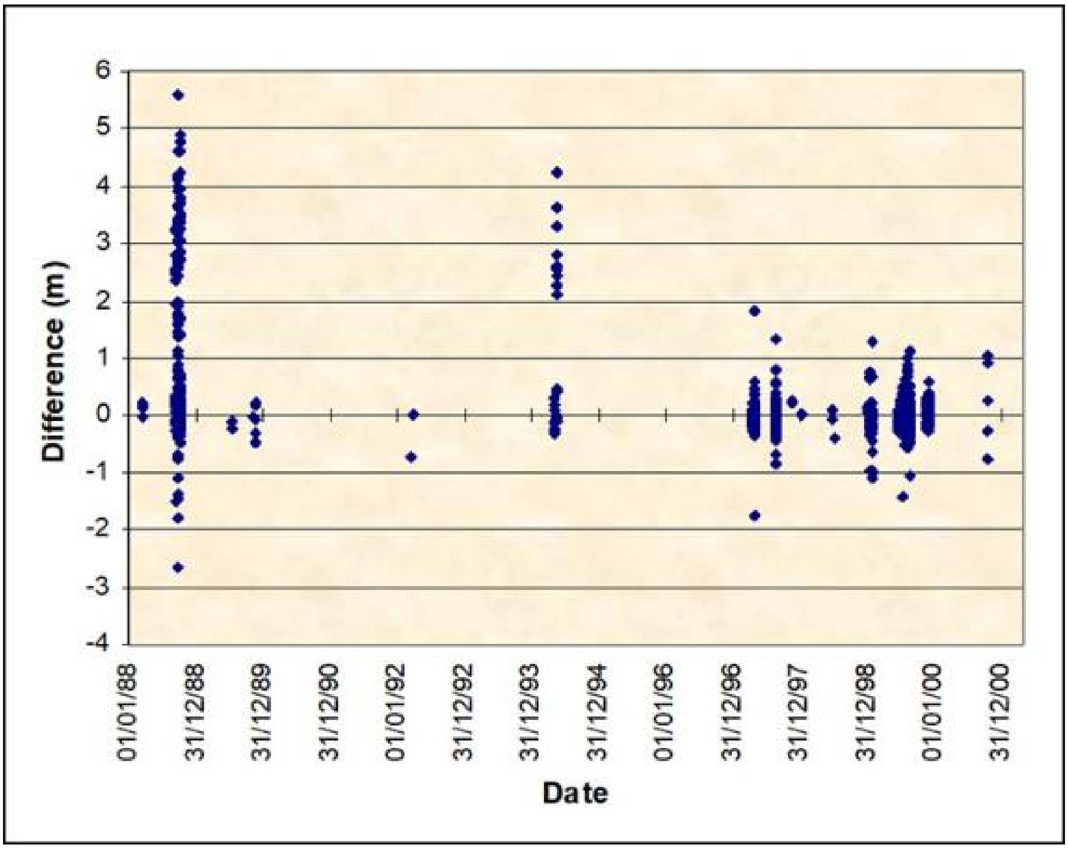

Figure 9-1: **Differences Between Drill Hole Collar Elevations and Topographic Wireframe by Date**

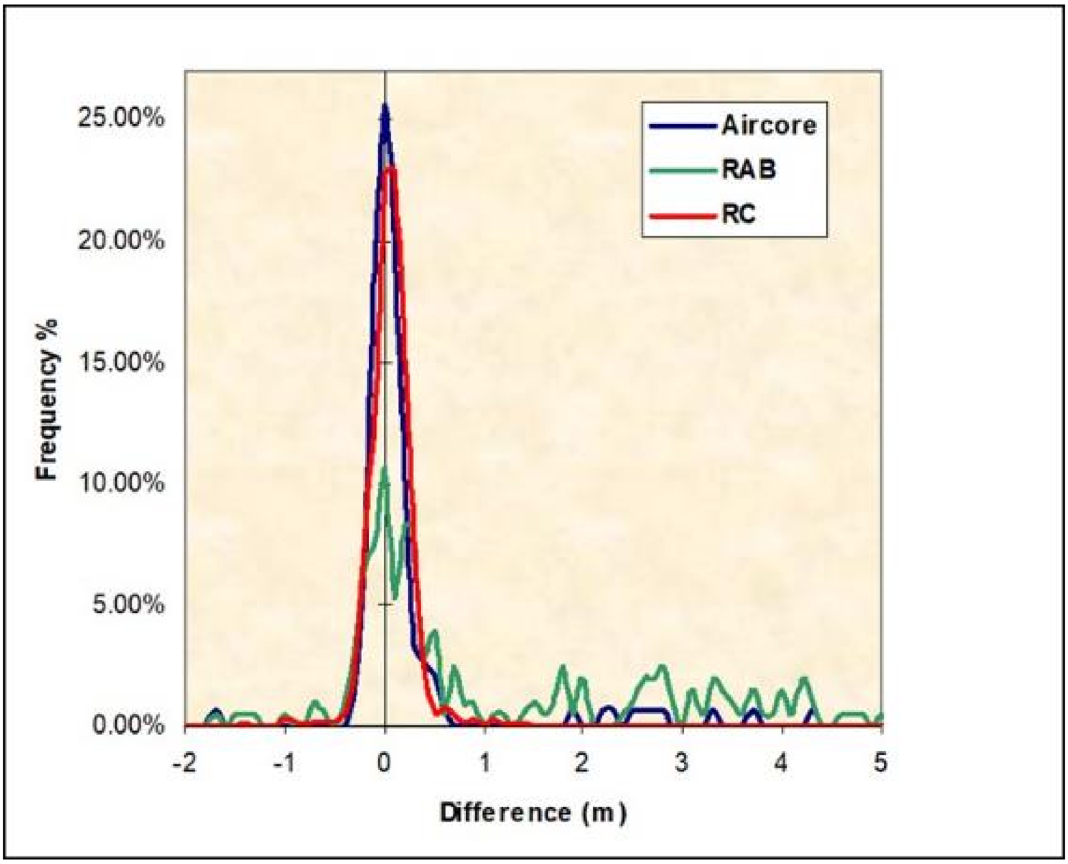

Figure 9-2: **Differences Between Drill Hole Collar Elevations and Topographic Wireframe by Hole Type**

In 2017, Clean TeQ Holdings Pty Ltd commissioned AAM Pty Limited (AAM) to carry out Airborne Laser Scanning and simultaneous photo imagery at Sunrise, as outlined in AAM's proposal (AAM, 2017[15]). This work covered various client-supplied polygons, including ML 1770, Westella Limestone area (ML 1769), the railway siding and the bore field, as well as Electrical and Infrastructure Corridors. The purpose of the work was to compile up-to-date terrain data for future development. Results are reported in (AAM, 2018[16])

This work was completed between 4 October 2017 and 4 January 2018, and included the following:

- ground control network for ground survey
- classified LiDAR ground point cloud in LASv1.2 format
- non-ground LiDAR point cloud in LASv1.2 format
- thinned LiDAR ground point cloud in LASv1.2 format
- 0.5m contours in DXF format
- 10cm GSD RBG orthophoto mosaic in ECW format
- 10cm GSD RBG tiles in ECW/TIF format
- 25cm GSD RBG orthophoto mosaic of corridors in ECW format
- 25cm GSD RBG tiles of corridors in ECW/TIF format.

All data was supplied in terms of GDA94 MGA Zone 55 and AHD (Ausgeoid09).

9.4 Regional Geophysical Surveys

A detailed aeromagnetic survey was flown by UTS Geophysics for Uranium Australia NL in July 1998. The survey is not open file, but data was provided to Clean TeQ after proof of ownership was shown to NSW Resources and Energy.

Details of the aeromagnetic survey flown over the Project are provided in Table 9-5.

Table 9-5: 1998 Aeromagnetic Survey

Survey Name	Sunrise Project
Contractor	UTS Geophysics
Client	Uranium Australia NL
Survey year	1998
Status	Confidential
Job number	A280
Methods	MAG DEM
Flight line spacing	50m
Flight line direction	090-270 degrees (east-west)
Mean terrain clearance	25 metres

In addition, a sub-audio magnetics (SAM) survey was conducted by GAP Geophysics in 2009, over the dunite complex. This produced ground magnetic and equivalent magnetometric resistivity data. A SAM report was located by Clean TeQ and data were resupplied by GAP Geophysics.

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[15] AAM Pty Limited, 2017. Updated Proposal for Syerston Nickel-Cobalt Project Aerial Survey, 7 August 2017

[16] AAM Pty Limited, 2018. Clean TeQ Holdings Ltd, Syerston Aerial Survey. Volume PRJ30146NOB01, Data Documentation. 5 February 2018

In 2017, Clean TeQ engaged Southern Geoscience Consultants (SGC) to perform an interpretation of the existing aeromagnetic data which covers the Project area. The aims of the interpretation were to identify structure and lithology within the Tout Intrusion. Additionally, data from a previous SAM survey has been located, processed and incorporated into the interpretation. Results of the interpretation are presented in SGC's report 3288 (Trunfull and Gilgallon, 2018[17]).

The interpretation has highlighted regional and local scale structures, magnetic zonation and possible sub-plumes within the intrusion. Subject to geochemical review by Nigel Brand of Geochemical Services Pty Ltd, some of the structures identified within the laterite ore zone should be targeted for increased density grade control drilling.

Neither the aeromagnetic data nor the SAM data could resolve microstructures on the scale of the required close-spaced grade control drilling. SGC recommends small field trials using alternative geophysical methods (ground penetrating radar, electromagnetic, microgravity etc) to determine if microstructures can be resolved.

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[17] Trunfull, J. & Gilgallon, K., 2018. Clean TeQ Sunrise Project Aeromagnetic Interpretation. Southern Geoscience Consultants Pty Ltd. Report SGC3288, January 2018 [Unpublished].

10 Drilling

10.1 Drilling Programmes 1986-2015

The numerous phases of drilling that occurred over the deposit during the period September 1986 to November 2015 were reviewed by McDonald Speijers and are summarised in Table 6-1. Overall drilling statistics and Project totals are presented at the end of this table. SRK Consulting also undertook a review of past drilling and reported that there were 1,733 holes totalling 61,752 m available as at 1 February 2016. Of those, 1,363 holes aggregating 49,900 m were judged to be acceptable for use in resource estimation. These are overwhelmingly RC holes, with a small proportion of AC holes (about 3% of the total) accepted only where there were substantial gaps in the RC drilling pattern. Drill hole locations are shown in Figure 10-1 and Figure 10-2.

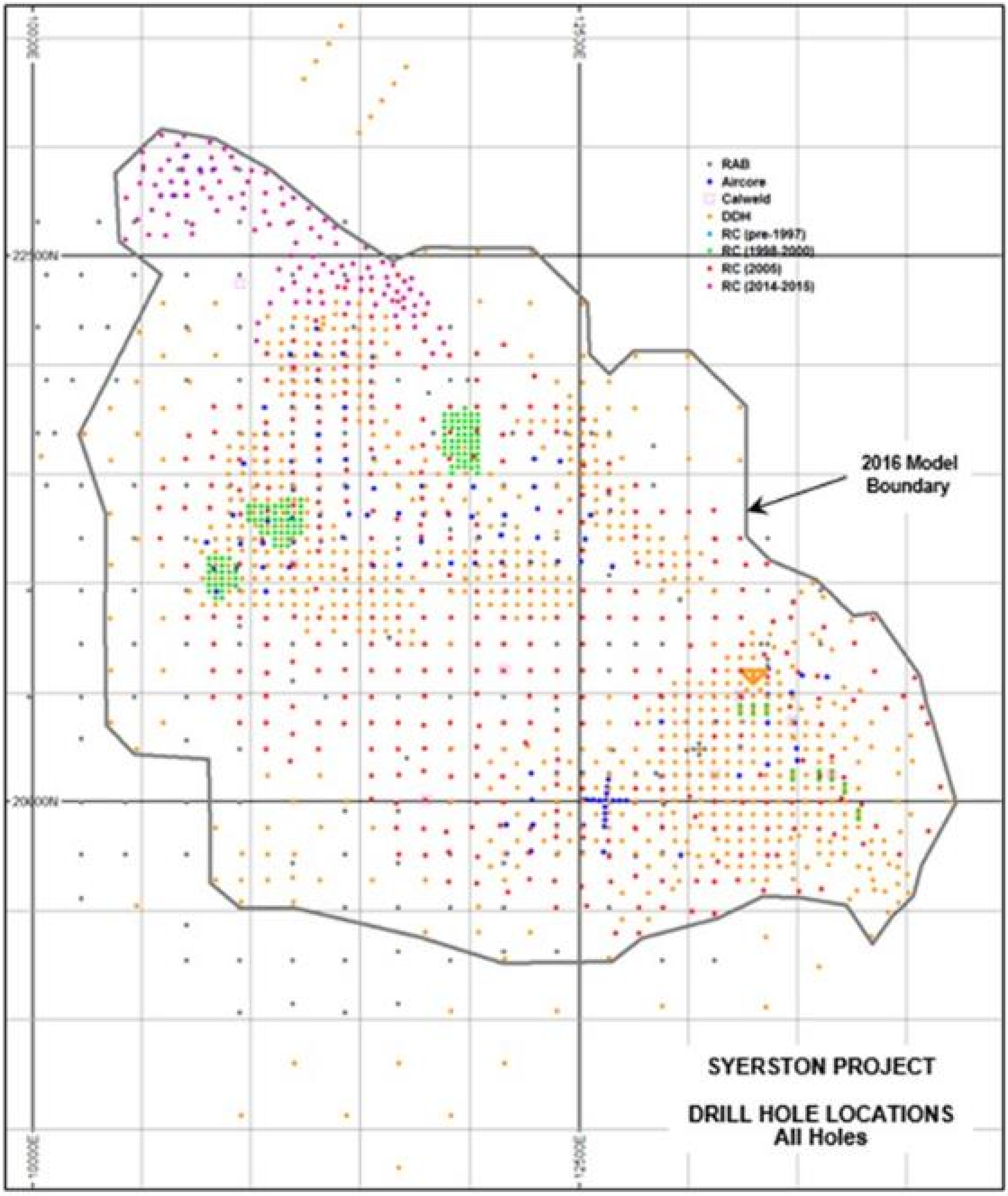

Figure 10-1: Drill Hole Location Plan - All Holes Drilled up to November 2015

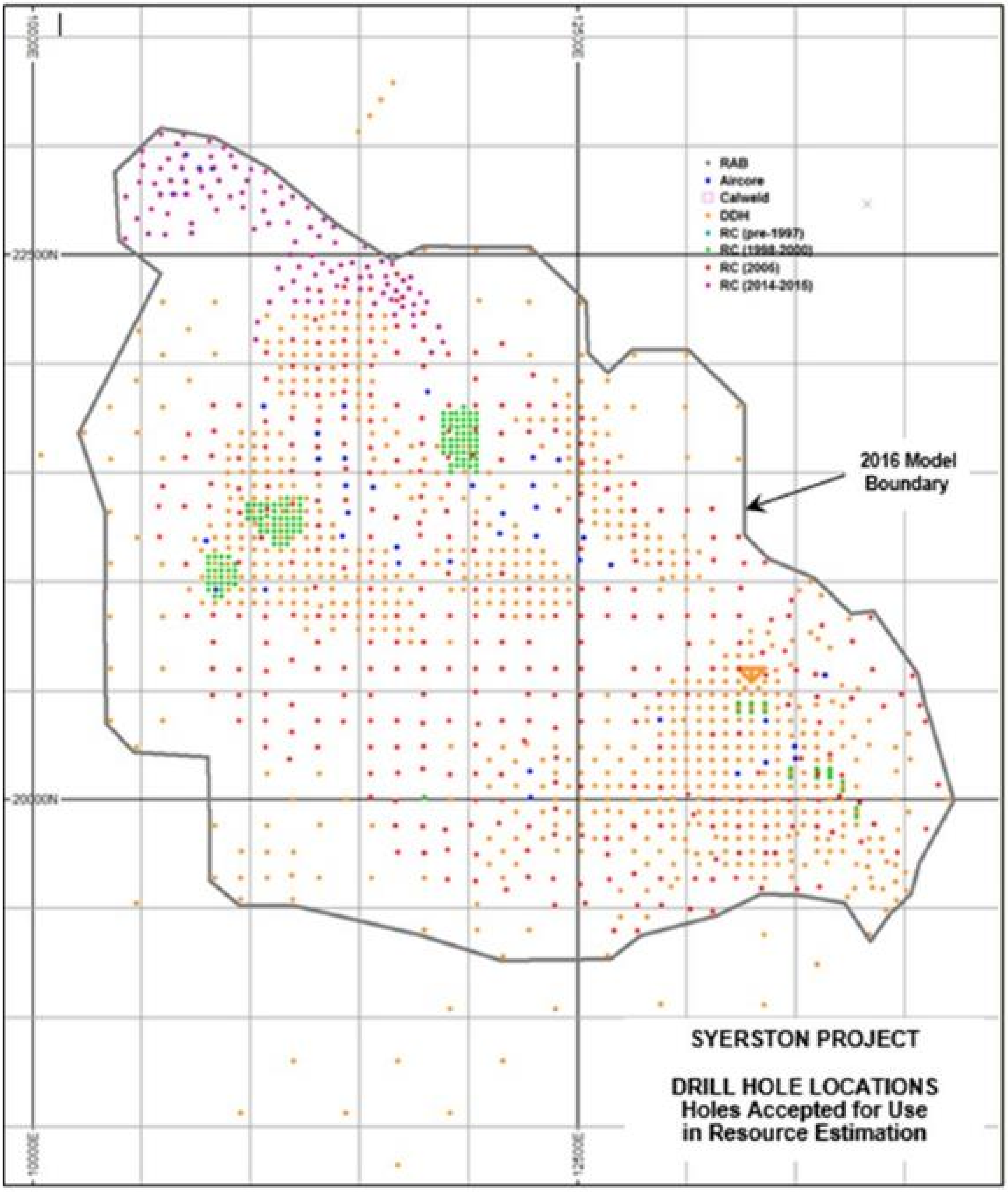

Figure 10-2: Drill Hole Location Plan - Holes Used in Resource Grade Estimation

Table 10-1: Summary of Drilling Programs up to November 2015

Time Period	Operator	Hole Type	Total			Used in 2017 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 platinum 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 and 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 and 22/03/99) and lacking details of the drilling contractor. However, co-ordinates are not the same as matching FR series numbers.
		RAB	P30 and 36	2	73	P30 and 36	0	0	Start and end dates and drilling contractor as above. No assay data. Same depths and within 20m in co-ordinates of FR30 and 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 platinum 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.
		RC	SRC001-340	341	14,149	SRC001-340	339	14,069	Infill drilling

Time Period	**Operator**	**Hole Type**	**Total**			**Used in 2017 Resource Model**			**Comments**
			Hole Number Range	**Number of Holes**	**Metres in Database**	**Hole Number Range**	**Number of Holes**	**Metres in Database**	
Aug 1997 - Aug 1998	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	25,870	SRC341-1076	724	25,710	Infill drilling; additional holes drilled after August 1999 are not reported in feasibility study documents. Note: SRC1061 to SRC1064 are excluded.
		DDH	SDD6-13	8	319.1	SDD6-13	0	0.0	PQ3 core for density and metallurgy tests; holes SDD6,8,9,10 and 12 were not assayed.
		Calweld	SCW1-9	9	234	SCW1-9	0	0	Large diameter holes; three holes in November 1999 were not reported in the feasibility study documents.
Feb 2005 - Mar 2005	Ivanplats Syerston	RC	SRC1077-1193	117	4,325	SRC1077-1193	117	4,325	Local 30x30m infill drilling.
			SRC1194-1225	32	1,594	SRC1194-1225	32	1,594	Infill drilling around elevated platinum intercepts.
			SRC1226-1251	26	833		0	0	Twinning previous Calweld and RC holes.
Aug 2014	Ivanplats Syerston	RC	SRC1263-1276	14	381	SRC1263-1276	14	381	Testing for peripheral scandium zones.
Apr 2015	Scandium21	RC	SRC1277-1310	34	944	SRC1277-1310	34	944	Infill drilling in areas of elevated scandium.
Nov 2015	Scandium21	RC	SRC1311-1368	58	1436	SRC1311-1368	58	1436	Infill drilling in areas of elevated scandium.
Project Totals		**RAB**		**207**	**6,891**	**-**	**0**	**0**	-
		Aircore		**148**	**4,275**	**-**	**45**	**1,441**	-
		RC		**1,354**	**49,532**	**-**	**1,318**	**48.459**	-
		DDH		**15**	**820**	**-**	**0**	**0**	-
		Calweld		**9**	**234**	**-**	**0**	**0**	-
		ALL		**1,733**	**61,752**	**-**	**1,363**	**49,900**	-

10.2 Drilling Programmes - 2016 – April 2018

Additional drill holes have been completed during 2016 and 2017 as follows. None of these holes have been used in the 2016 or 2017 Resource Estimations.

- SRC1369 to SRC1383 - 23 February 2016 to 27 February 2016
- SRC1384 to SRC1417 (sterilisation drill holes) - 4 July 2017 to 11 July 2017
- SRC1418 to SRC1427 (RC twin holes) - 1 September 2017 to 4 September 2017
- SDD014 to SDD021 (Diamond core holes drilled adjacent to twin holes) - 6 October 2017 to 7 November 2017
- SRC1428 to SRC1551 – Close-spaced (20mx20m) RC holes were drilled in four separate locations in the proposed mining area: Area A (southeast), Area B (northeast), Area C (southwest), Area D (northwest). – 18 January 2018 to 22 February 2018
- SRC1552 was drilled on 22 February 2018 adjacent to existing twin holes to provide samples for metallurgical testwork on platinum.

10.3 Drilling methods

The following discussion of drill hole methods is largely based on McDonald Speijers 2005 and McDonald Speijers 2016.

10.3.1 RAB Drilling

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.

10.3.2 Reverse Circulation and Aircore Drilling

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. These were located 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. All drilling was above the water table and samples were generally dry to slightly moist. Most of the holes that were accepted for use in resource estimation were of this type. Similar rigs were used for the 2014-2015 RC drilling.

RC holes provided the main basis for recent resource estimates. Inclusion of more AC holes would provide better data support for resource estimates, mainly in mineralised zones above the SGZ. However, the collar coordinates of the AC holes could not be validated with enough accuracy for use in measured resource estimation. In addition, there were indications of a possible sampling bias between AC and RC holes.

10.3.3 Diamond Drilling

Very little diamond core drilling has been carried out. Two holes were drilled in 1992 to test for bedrock platinum mineralisation. These are of little relevance to the lateritic resource. In 1997, 1998 and 1999, a total of 13 shallow, vertical, diamond core holes were drilled at PQ3 size (with core diameter around 83 mm), using triple tube equipment and split tube core barrels.

These holes were intended mainly to provide undisturbed samples for density measurements and for metallurgical testwork, but eight of them were systematically sampled and assayed.

Core recovery was erratic, with logs and photographs indicating that it tended to be either near 100% or near zero. On average, recovery was approximately 85% through the main mineralised zones.

The low average core recovery was too low to have a high level of confidence in the assay results and they were therefore not an acceptable basis for measured resource estimation. Consequently, diamond drill holes were excluded from resource estimation. In most cases they were drilled near pre-existing RC holes which provided suitable assay data.

10.3.4 Calweld Drilling

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

The rig used a 750 mm diameter bucket fitted with a hinged base and set with cutting teeth extending 10 mm outside the bucket, producing a total hole diameter of about 770 mm. The bucket was driven by a boom-mounted chuck using 6 m telescopic rods (a second 6 m length extending from inside the first rod). The bucket was about 1 m deep, but apparently drilled only about 0.3-0.4 m at a time before being withdrawn and emptied by opening the base.

A good photographic record of this drilling exists, which indicates that the holes were quite regular and smooth-sided.

In most cases the rig was unable to completely penetrate the SGZ, so the average hole depth was only about 26 m.

The rig seems to have achieved almost total recovery and should have provided good quality bulk samples for the intervals that could be penetrated. The inability of these holes to fully penetrate the SGZ, together with the small number of holes, makes the Calweld drill holes of very limited value for resource grade estimation purposes and they were excluded from use. However, where they were properly bulk sampled they provide a good check on other drilling methods. They also provided valuable bulk density information.

10.4 Drilling patterns and hole orientation

By 2000, a drilling pattern of approximately 120 m x 120 m had been completed over much of the resource area, with infill drilling to approximately 60 m 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 m x 30 m was conducted in the areas of some year one pit designs generated during the 2000 feasibility study. These had focused on areas of unusually high cobalt grades in the resource model of the time. The closer spaced drilling was intended to either confirm the continuity of high cobalt values or to better constrain their volumetric influence.

With the exception of two diamond drill holes from 1991-1992, all holes have been vertical, with an average depth of approximately 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 platinum grades.

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

10.5 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 AC 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 AC and RC holes for both nickel and cobalt. On average, AC grades tended to be lower by around 10% relative for nickel and 15% relative for cobalt.

The most reliable measurements of in situ nickel and cobalt grades came from several large diameter Calweld holes which 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 AC 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 approximately 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 (see Table 10-2).

At the same time, the old RC holes located near the Calweld holes were also twinned with collar separations of approximately 1.5 m. The results were also satisfactory for both nickel and cobalt, with 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

Criteria	Pairs	Total Metres	Grade Nickel (%)	Grade Cobalt (%)	Total Metres	Grade Nickel (%)	Grade Cobalt (%)
		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 cobalt grade comparisons between bulk sampled Calweld holes and the nearest existing RC holes at the time were due primarily to inherent, short-range, cobalt grade variability. This has implications for future grade control requirements should a nickel and cobalt project proceed.

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

10.6 Drill hole sampling

Most of the 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. Full details are provided in McDonald Speijers (2005).

10.6.1 Sampling procedures

10.6.1.1 Aircore holes

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

10.6.1.2 RC holes: 1997 - 2000

Sampling procedures for most 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 1m intervals. Total sample was collected via rig-mounted cyclones and riffle split to about 2-4kg, using three tier multistage 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. McDonald Speijers considered that 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".

10.6.1.3 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 three-tier riffle splitter. 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 larger (polyweave) 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 like 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.

10.6.1.4 RC holes: 2014 - 2015

According to OreWin[18], 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 splitter 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 like those used during previous RC drilling programs conducted since 1997, except for the initial use of 2 m composite sample intervals.

Procedures in 2015 were supposed to have been like 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.

10.6.1.5 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 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.

10.6.1.6 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 approximately 10kg 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.

10.6.2 Sample recovery

10.6.2.1 Aircore holes

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

10.6.2.2 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.

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[18] OreWin, 2016. Scandium21 Pty Ltd, Syerston Project Feasibility Study. Scandium Mineral Resource. Job No. 15017, May 2016 [Unpublished].

Weighing RC samples is often good practice, but McDonald Speijers was 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 (see 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. Results are discussed in Snowden (2000)[19].

McDonald Speijers stated that, in their 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 Reverse Circulation 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 cobalt and possibly a modest positive bias for nickel 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.

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[19] Snowden Mining Industry Consultants Pty Ltd, 2000. Syerston RC Drilling Sample Recovery Review. Report to SNCL, January 2000 [Unpublished.

For current drilling (2018) the standard procedures for drilling and sampling are documented in Clean TeQ 2018a[20]

10.6.2.3 Diamond drill holes

For diamond core holes drilled up to 2000, core recovery was reportedly erratic. Logs and photographs indicated that it tended to be either near 100% or near zero. On average, recovery was estimated to be about 85% through the main mineralised zones.

Core recovery which was recorded for the eight diamond core holes which were drilled in late 2017 was measured and summarised as follows: (see Table 10-4).

Table 10-4: Summary of Diamond Core Recoveries For 2017 Drill Holes SDD014 to SDD021

Hole Number	Final Depth	Recovered Core (m)	Core Loss (m)	Core Recovery (%)
SDD014	69.0	63.25	5.75	91.67
SDD015	59.7	30.48	29.22	51.06
SDD016	62.0	50.1	11.9	80.81
SDD017	66.6	45.0	21.6	67.57
SDD018	71.0	67.4	3.60	94.93
SDD019	41.3	28.87	12.43	69.90
SDD020	56.2	44.95	11.25	79.98
SDD021	52.4	47.53	4.87	90.71

10.6.2.4 Calweld holes

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

10.6.3 Drilling Statistics

Overall drilling statistics for drill holes included in the 2017 resource estimate are shown at the bottom of the main drilling table (see Table 6-1 above).

10.1 Bulk density

10.1.1 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 were used, as shown in Table 10-5.

10.1.2 Downhole gamma logging

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 10-5.

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[20] Clean TeQ, 2018a. Sunrise Nickel-Cobalt-Scandium Project, New South Wales Australia. 2017 Geological Protocols and Procedures Manual, Version 1.0 [Unpublished].

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.

10.1.3 Calweld holes

McDonald Speijers located 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.

10.1.4 Summary

The average dry bulk density values indicated by the various methods are compared in Table 10-5.

Table 10-5: Comparison of average dry density values

Zone	DD Core Volumetric Measurements (Five Holes)	DD Core Water Immersion (One Hole)	Gamma/Logging (136 Holes With Dry Densities)	Calweld Holes (Six 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.

10.1.5 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 10-6.

Table 10-6: 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
TZI	>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).

10.2 Geological Logging of RC Chips and Diamond Core

Geological logging has been carried out in accordance with logging protocols and procedures which were in place at the time of drilling. Logging codes have evolved with the project, with the most recent version being based on 'Logging Methods and Codes' compiled under the title Scandium21 Pty Ltd – Sunrise Project. Logging of recent diamond holes and current RC drilling is expected to necessitate the addition of extra codes as new minerals and rock types are identified.

10.3 Geotechnical Logging of Core

There are no records of geotechnical logging on any of the diamond core holes which were drilled in the following programmes:

- FPD1 and FPD2 – These holes were drilled by the Noble-Poseidon JV between September 1991 and September 1992. These holes were drilled into fresh pyroxenite in the north-western part of the deposit
- SDD1 to SDD5 – Drilled by Uranium Australia between August 1997 and August 1998
- SDD6 to SDD13 – Drilled by Black Range Minerals between August 1998 and October 2000.

Recently-drilled diamond core holes SDD014 to SDD021 (drilled in late 2017) had rock quality designation (RQD) measurements made on all core.

Holes SDD014 to SDD018 are located at the eastern end of the deposit and SDD019 to SDD021 are close to the western end. The results are summarised in Table 10-7.

Table 10-7: Summary of RQD Measurements For 2017 Drill Holes SDD014 to SDD021

Hole Number	RQD-Measured Core	RQD Percentage
SDD014	16.32	25.80
SDD015	3.97	13.02
SDD016	10.78	21.52
SDD017	10.09	22.42
SDD018	15.21	22.57
SDD019	4.10	14.20
SDD020	5.40	12.01
SDD021	8.99	18.91

RQD is a rough measure of the degree of jointing or fracturing in a rock mass. Rock with a RQD measurement of below 50% is designated as low quality. The RQD measurements on core from these diamond holes indicate that the rock is of extremely low quality.

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, Ultra Trace Analytical Laboratories (Ultra Trace) based in 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 the ALS facilities in Orange, New South Wales.

In 1999, significantly mineralised intervals (>0.4% nickel) from previous AC holes and from RC holes up to SRC340 (which were drilled between 1995 and 1998), were re-assayed at Ultra Trace. The intervals involved were apparently selected on the basis of a nickel equivalent value >0.25%, apparently using the formula NiEQ%= Ni% + (Co%x3.64).

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 flexible intermediate bulk container (FIBC) 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)
- FIBC 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 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 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 FIBC sacks
- 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 FIBC sack 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 were conducted in 1999 and 2005.

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
- 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 (see Section 11.6). The average grades reported were also like 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 1m 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 approximately 1kg, 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	0–43x	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:

- total sample was pulverised to 90% -75 µm, with mills cleaned before and after the batch with barren quartz washes assaying
- for platinum, gold: 50g fire assay, lead collection, inductively coupled plasma (ICP) finish
- for nickel, cobalt, magnesium, iron, chromium, manganese, aluminium, calcium, scandium, zinc, copper, arsenic, mixed four acid digest with inductively coupled plasma optical 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 silicon: 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 Ultra Trace 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 assaying procedures used by ALS would have been as detailed in the following sections.

Table 11-2: Assaying Procedures Pre-1999

Element	Assaying Procedures Comments
Platinum	Fire assay using a 50g charge, with lead collection and an atomic absorption spectroscopy (AAS) finish (method PM217). McDonald Speijers found mention of a very limited amount of screen fire assaying for platinum 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 platinum assays might be affected by coarse particles.
Gold	The drill hole database indicated that gold assays from this period were obtained by aqua regia digest and AAS determination. However, it would seem more logical for the gold assays to have been obtained as part of the fire assay process. McDonald Speijers suspected that the database was misleading in this regard. Gold values are generally very low and this is not a significant matter.
Nickel. Cobalt. Copper. Chromium. Manganese.	These elements were determined using a perchloric acid digest of a 0.25g portion of sample pulp, with an AAS finish (method G001). Nickel, cobalt and chromium were assayed routinely with manganese values for most samples from AC holes. Only a very small number of copper assays are recorded.
Magnesium. Iron.	It appears that selected sample pulps were assayed retrospectively for these elements. A 0.25g 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 scandium, aluminium and silicon 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 Ultra Trace prior to 2005 were as detailed in the following sections.

Table 11-3: Assaying Procedures Pre-2005

Element	Assaying Procedures Comments
Platinum. Gold.	Digestion of a nominal 40g 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 platinum in parts of this deposit).

Element	Assaying Procedures Comments
Nickel. Cobalt. Chromium. Manganese. Magnesium. Calcium. Aluminium. Iron. Scandium. Zinc. Arsenic. Copper	Digestion to dryness of 0.3g 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 ICP-OES 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 ten days). McDonald Speijers considered that reported moisture values were unlikely to provide a reliable indication of in situ moisture contents and stated that they would not include the moisture values in a resource model.

11.4.3 Ultratrace (2005)

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

- platinum and gold were determined by fire assay rather than aqua regia digest. The method used was normal lead collection fire assaying of a 40g 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 Ultra Trace results for all elements, with the possible exceptions of platinum and gold.

11.4.4 ALS 2014 - 2015

For the 2014-2015 analysis, an aliquot of 0.25g 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-OES.

11.4.5 Comparison between assaying methods

A substantial number of AC and RC holes originally assayed at ALS were subsequently re-assayed at Ultra Trace. In effect, this provided a substantial set of inter laboratory check assay data.

The different analytical methods used at ALS (perchloric digest, AAS finish) and at Ultra Trace (four acid digest, ICP-OES finish) produced very similar results for nickel and cobalt. Only chromium values showed a substantial difference.

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 did not review internal laboratory QC results. External QC checks, preferably totally blind to the laboratory, were regarded as more significant and McDonald Speijers 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 like 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 Ultra Trace and submitted to Genalysis in Maddington, Western Australia, for check assaying for:

- nickel
- cobalt
- copper
- iron
- manganese
- chromium
- scandium
- calcium
- aluminium
- magnesium
- zinc.

using a four-acid digest with an ICP-OES finish (like the analytical method used by Ultra Trace).

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 AC and RC holes that were initially assayed by ALS between 1995 and 1998, with pulps from mineralised intervals subsequently being re-assayed by Ultra Trace 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 AC 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 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 concluded that the final products were packaged into 25g sealed sachets, ready for use.

Gannett apparently provided recommended nickel, cobalt, manganese, iron, magnesium and aluminium values for each standard based on 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 11-4. These were extracted from the Drill Hole Standards table (tblDH standards) in the drill hole database Syerston_DB.mdb. Each of these standards was apparently submitted under four different field standard identification numbers.

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 nickel mineralisation were obtained from Geostats Pty Ltd, White Gum Valley, Western Australia (see Table 11-4). 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 platinum, so none of the standards could provide any check on the accuracy of platinum 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 scandium assay results. Nickel and cobalt grades of that standard were too low to provide useful data.

Table 11-4: **Standard Sample Recommended Values and Provenance**

Standard Number	Field Standard Number	Prepared By	Provenance	Nickel (ppm)	Cobalt (ppm)
SYS1	S4, S8, S12, S20	Gannett	Clean TeQ Sunrise mineralisation	6260	740
SYS2	S1, S6, S10, S16	Gannett	Clean TeQ Sunrise mineralisation	8,980	2,150
SYS3	S11, S14, S15, S19	Gannett	Clean TeQ Sunrise mineralisation	10,120	3,980
SYS4	S3, S7, S13, S17	Gannett	Clean TeQ Sunrise mineralisation	7,480	2,660
SYS5	S2, S8, S9, S18	Gannett	Clean TeQ Sunrise mineralisation	9,330	3,540
GBM902-2	GBM902-2	Geostats Pty Ltd	Nickel laterite ore, Queensland	3,014	975
GBM901-1	GBM901-1	Geostats Pty Ltd	Nickel laterite, Kalgoorlie Region	8,037	1,346
GBM901-2C	GBM901-2C	Geostats Pty Ltd	Nickel laterite ore, Eastern Goldfields	8,830	314
GBM302-8	GBM302-8	Geostats Pty Ltd	Nickel laterite, Eastern Goldfields	10,775	483
GBM900-9	GBM900-9	Geostats Pty Ltd	Nickel laterite, Western Australia	11,615	567

McDonald Speijers understood 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 up to 2016

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 observed for similar datasets.

11.6.1.1 1999 - 2000 results

McDonald Speijers (2005) state that they 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 from Ultra Trace, as specified in the standard operating procedures of the time.

Summary statistics for duplicate samples submitted to Ultra Trace in 1999-2000 are shown in Table 11-5, Figure 11-1 and Figure 11-2.

11.6.1.2 2005 results

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

Results were like those reported from 1999-2000, except that the proportion of outliers in the results for nickel and cobalt were considerable higher than normal. The results are summarised in Table 11-6, Figure 11-3 and Figure 11-4.

11.6.1.3 Discussion

Results for nickel 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 cobalt, 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 suspected 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 cobalt were within satisfactory limits and similar over time.

Table 11-5: Duplicate Samples Summary Statistics (1999-2000)

Element		All Data				Probable Outliers Excluded				
		Original	Duplicate	Mean	Relative Difference	Proportion Excluded	Original	Duplicate	Mean	Relative Difference
Nickel	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%**
Cobalt	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%**

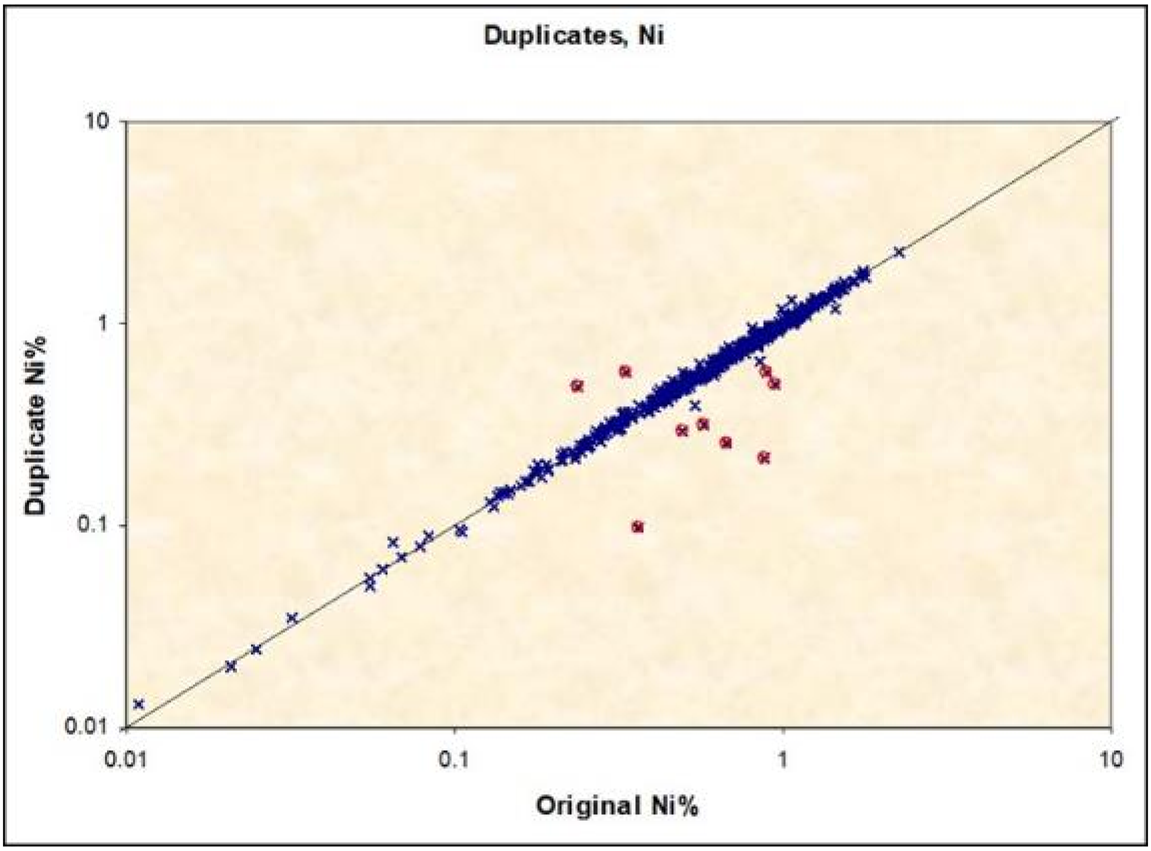

Figure 11-1: Duplicate Samples 1999-2000 (Nickel)

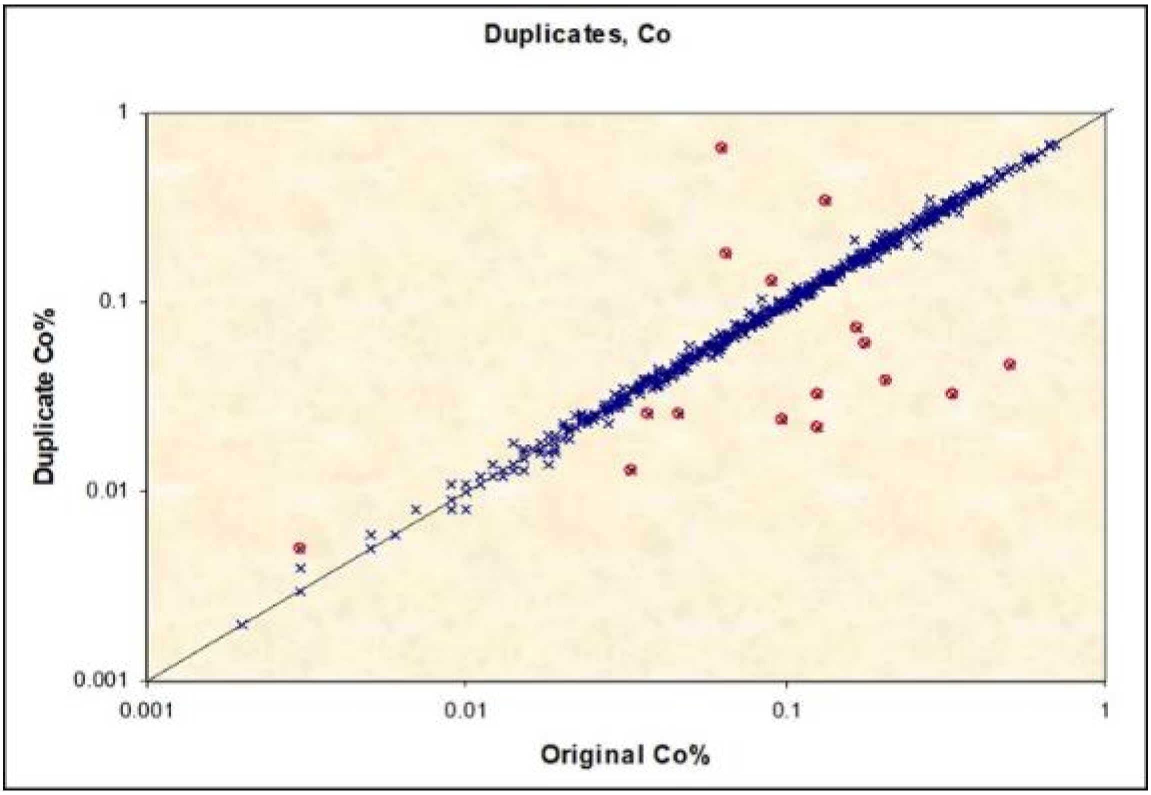

Figure 11-2: Duplicate Samples 1999-2000 (Cobalt)

Table 11-6: Duplicate Samples Summary Statistics (2005)

Element		All Data				Probable Outliers Excluded				
		Original	**Duplicate**	**Mean**	**Relative Difference**	**Proportion Excluded**	**Original**	**Duplicate**	**Mean**	**Relative Difference**
Nickel	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% Confidence Level)				**19.2%**					**6.1%**
Cobalt	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 (at 90% Confidence Level)				**35.7%**					**8.3%**

Precision levels for other major elements were in the range of about ±7-20%. The worst precision limits were for magnesium and manganese in 2005 which were around ±14% and ±20% respectively. Otherwise, they were generally within about ±10-12%, which should be adequate.

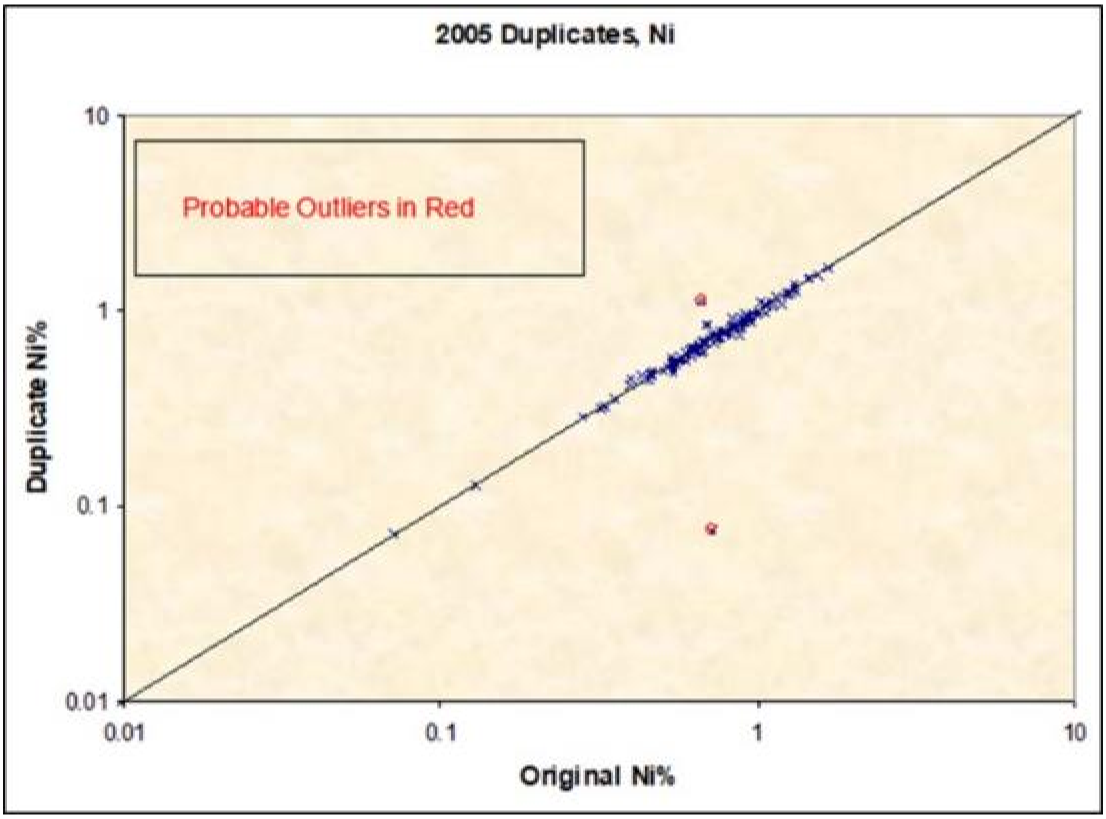

Figure 11-3: Duplicate Samples 2005 (Nickel)

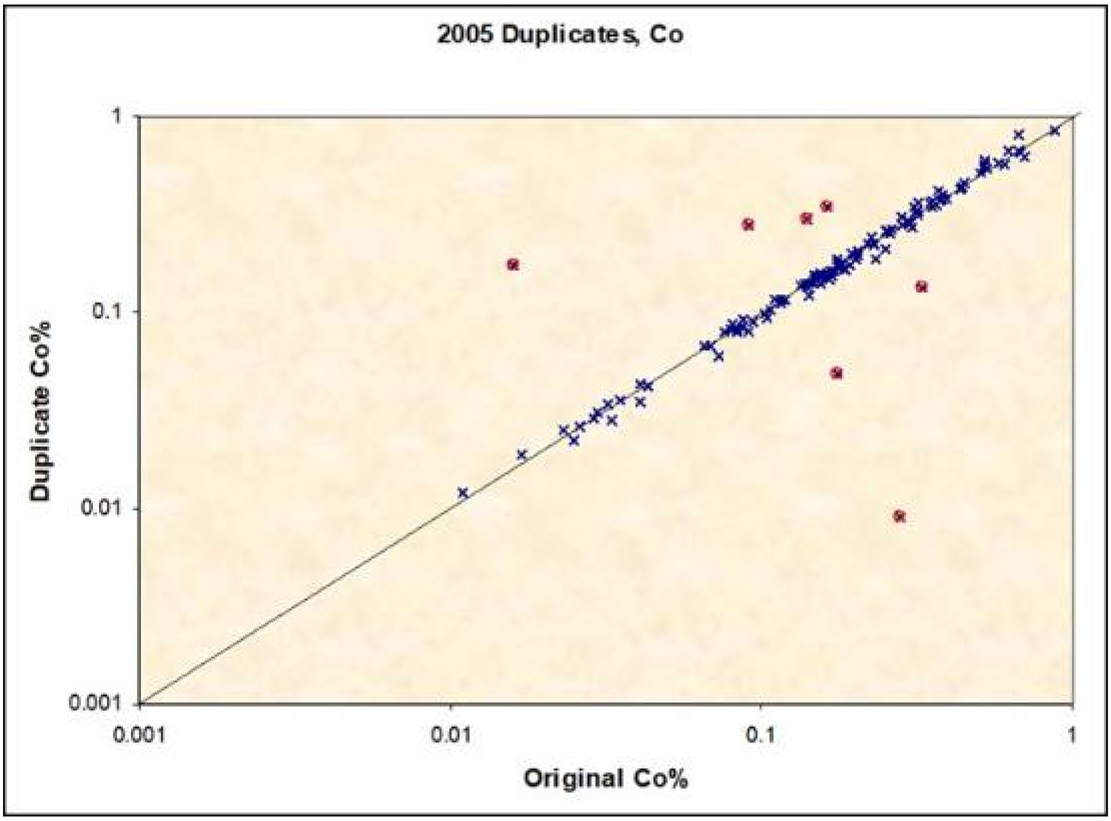

Figure 11-4: Duplicate Samples 2005 (Cobalt)

11.6.1.4 2014 - 2015

McDonald Speijers obtained results compiled by OreWin Pty Ltd[21] for 17 duplicate samples from 2m intervals and 88 duplicate samples from 1m 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% nickel and <0.01% cobalt). When these were excluded, biases were no longer evident (see Figure 11-5).

McDonald Speijers understood that duplicates were collected by spear sampling the bagged reject, although McDonald Speijers was not 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 Ultra Trace or Genalysis as the check laboratory, specifying the same analytical method they used in 2005, to retain a link to assaying procedures during previous drilling programs that provided the bulk of the assay database.

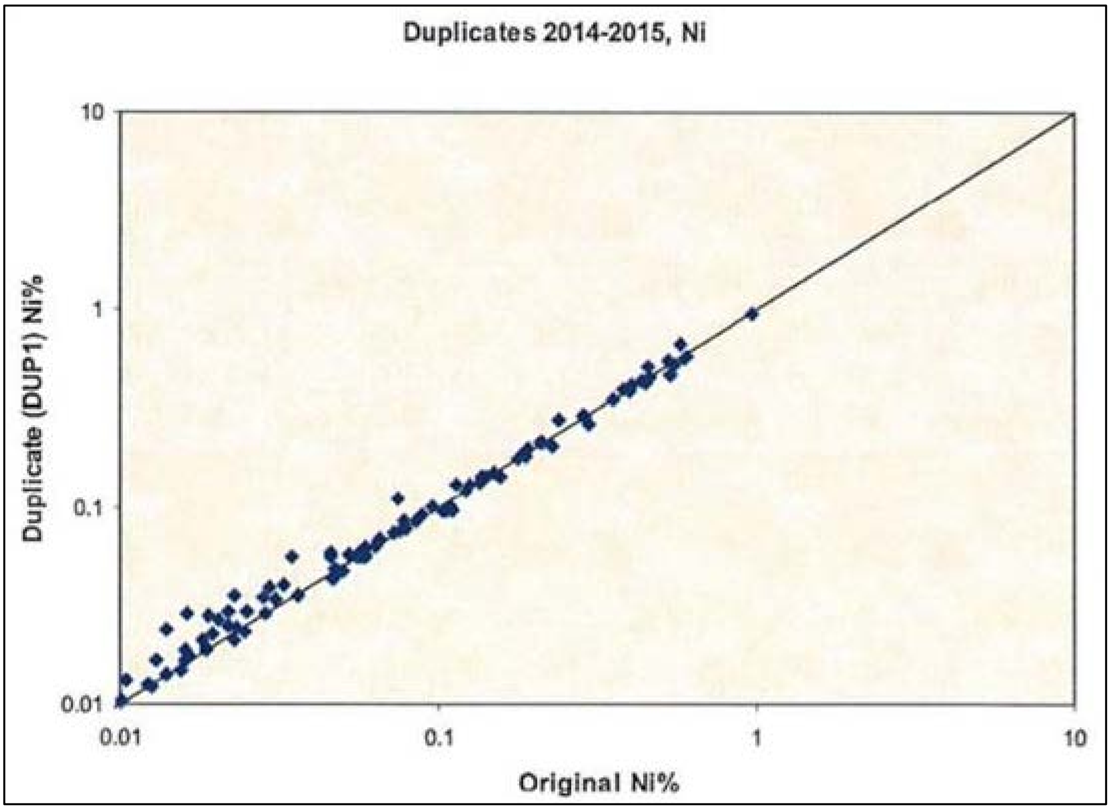

Figure 11-5: Duplicate Samples 2014-15 Nickel

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[21] Independent mining consultants, based in Adelaide, South Australia.

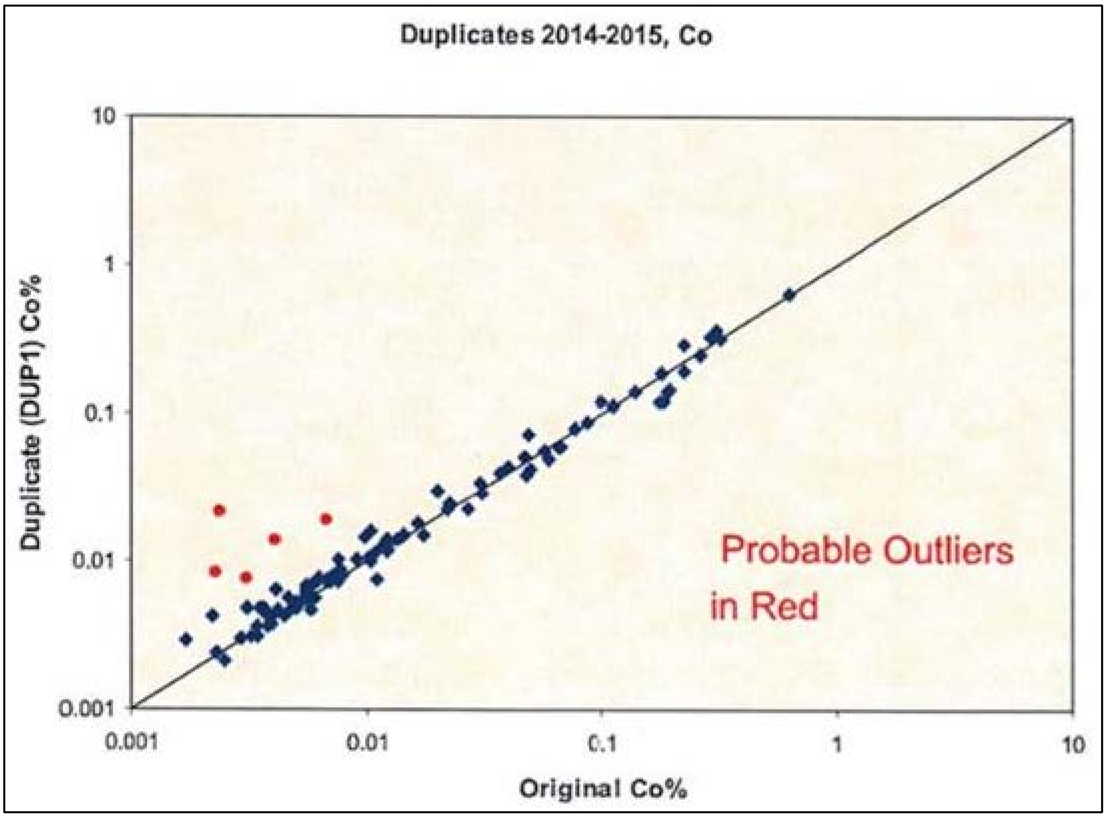

Figure 11-6: Duplicate Samples 2014-15 Cobalt

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 Ultra Trace 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 SRK Consulting stated that in their experience, the mean relative differences between reputable laboratories may fall in the range of about ±5%, although they should preferably be within ±2-3%.

11.6.2.1 Pre-1999 results

McDonald Speijers extracted 4,491 samples from the database that had nickel assays from both ALS and Ultra Trace and treated these as effectively being a large check assay dataset. There were similar amounts of data for cobalt, magnesium, chromium and manganese, more limited data for iron, magnesium and calcium and small datasets for a few other elements. There was no comparative assay information for platinum. The results from this period are summarised in Table 11-7.

11.6.2.2 1999 - 2000 results

McDonald Speijers obtained results for 785 samples assayed by both Ultra Trace and Genalysis for:

- nickel
- cobalt
- copper
- iron
- manganese
- chromium

- scandium
- calcium
- aluminium
- magnesium
- zinc.

in the 1999-2000 period.

These were contained in a spreadsheet (Genalysis Checks_Graphs.xls) in an old SLA archive directory provided to SRK Consulting 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 platinum assays. Summary statistics for this period are shown in Table 11-8.

11.6.2.3 2005 results

In 2005, a total of 231 original sample pulps were retrieved from Ultra Trace and submitted to Genalysis for check assaying for:

- nickel
- cobalt
- copper
- iron
- manganese
- chromium
- calcium
- aluminium
- magnesium
- zinc.

The results are summarised Table 11-7 to Table 11-9 and Figure 11-7 to Figure 11-12.

11.6.2.4 Discussion

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

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

Overall, the check assay results indicated that cobalt assays have been accurate.

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.

Table 11-7: Check Assays (Pre-1999) ALS vs Ultra Trace Summary Statistics

Element		All Data				Probable Outliers Excluded				
		ALS	Ultra Trace	Mean	Relative Difference	Proportion Excluded	ALS	Ultra Trace	Mean	Relative Difference
Nickel (%)	Pairs	4,491	4,491	4,491	4,491	0.3%	4,476	4,476	4,476	4,476
	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%
Cobalt (%)	Pairs	4,448	4,448	4,448	4,448	0.2%	4,439	4,439	4,439	4,439
	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 11-8: Check Assays (1999-2000) Genalysis vs Ultra Trace Summary Statistics

Element		All Data				Probable Outliers Excluded				
		Ultra Trace	Genalysis	Mean	Relative Difference	Proportion Excluded	Ultra Trace	Genalysis	Mean	Relative Difference
Nickel (%)	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%
Cobalt (%)	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 11-9: Check Assays (2005) Genalysis vs Ultra Trace Summary Statistics

Element		All Data				Probable Outliers Excluded				
		Ultra Trace	Genalysis	Mean	Relative Difference	Proportion Excluded	Ultra Trace	Genalysis	Mean	Relative Difference
Nickel (%)	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%
Cobalt (%)	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%

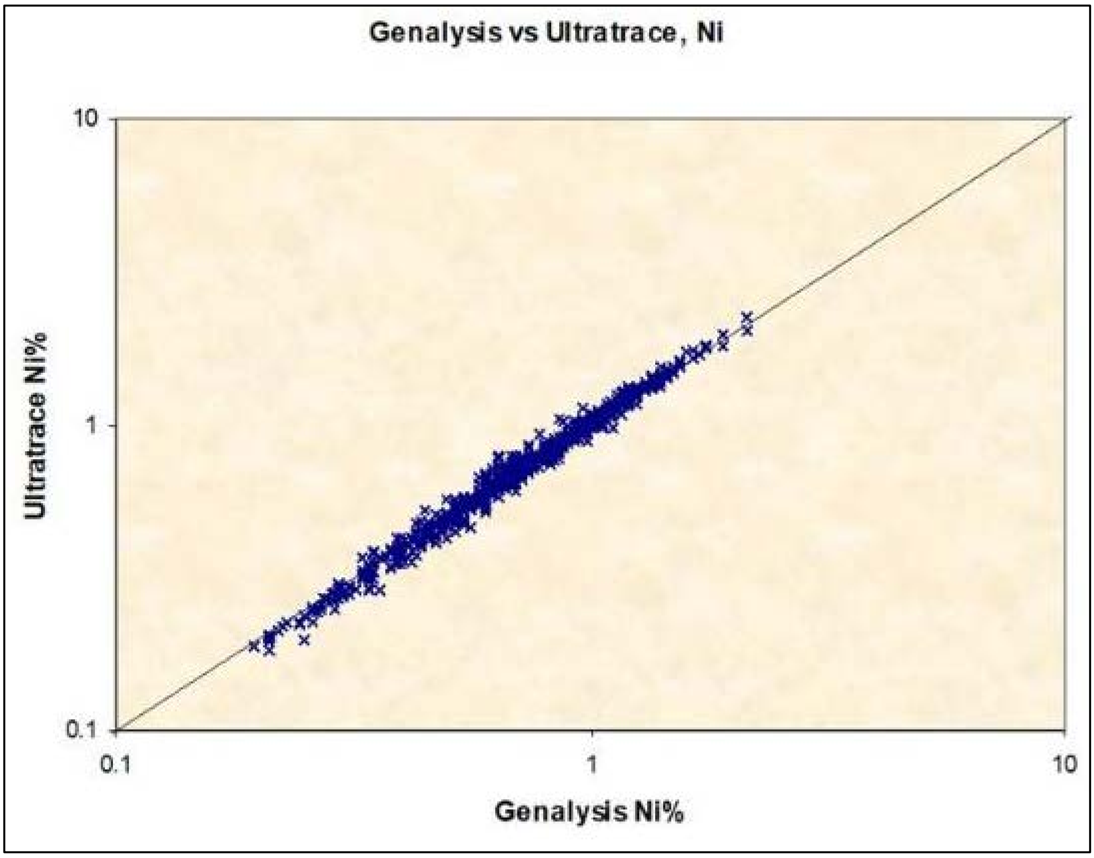

Figure 11-7: Check Assay Scatterplots Genalysis vs Ultra Trace, Nickel

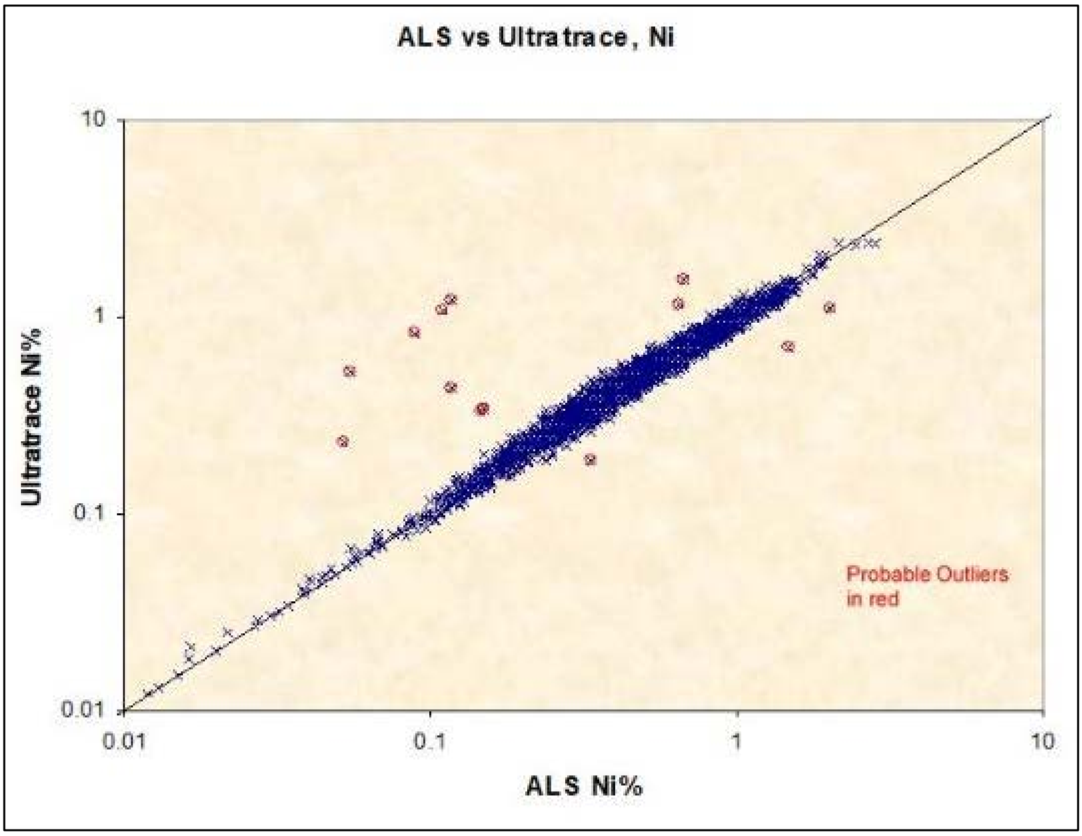

Figure 11-8: Check Assay Scatterplots ALS vs Ultra Trace, Nickel

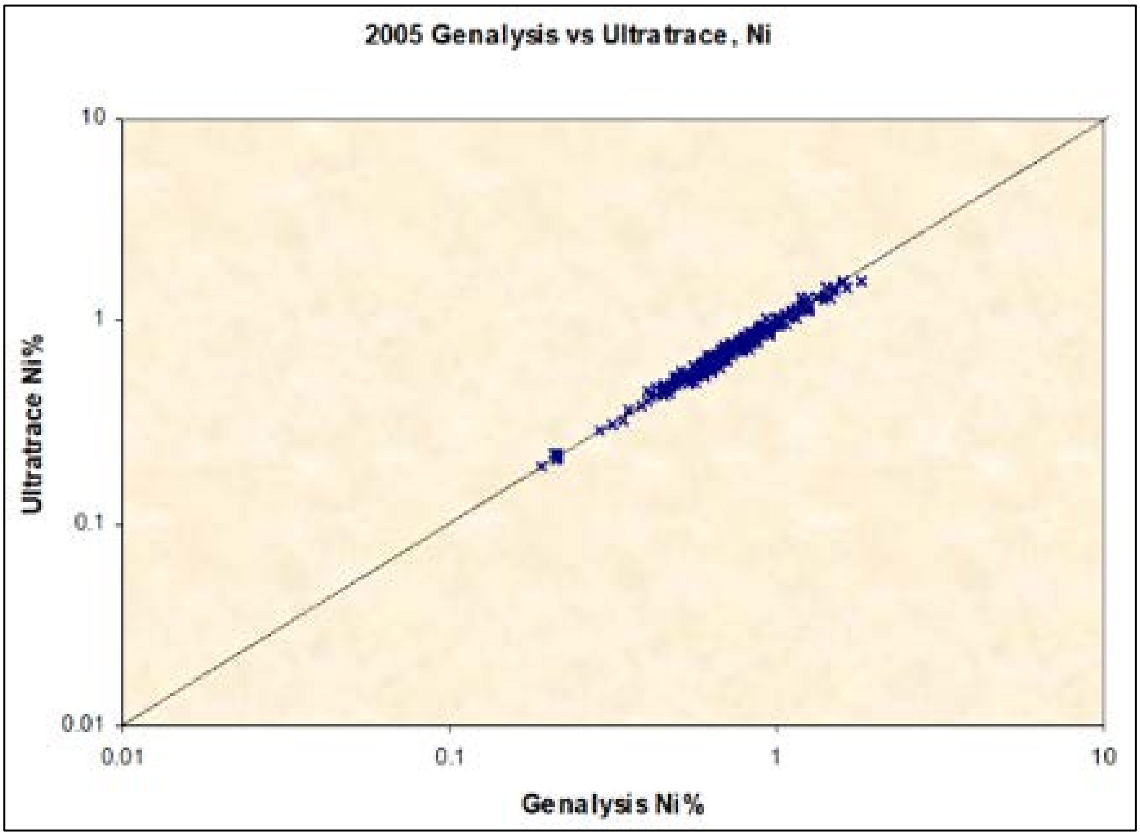

Figure 11-9: Check Assay Scatterplots 2005 Genalysis vs Ultra Trace, Nickel

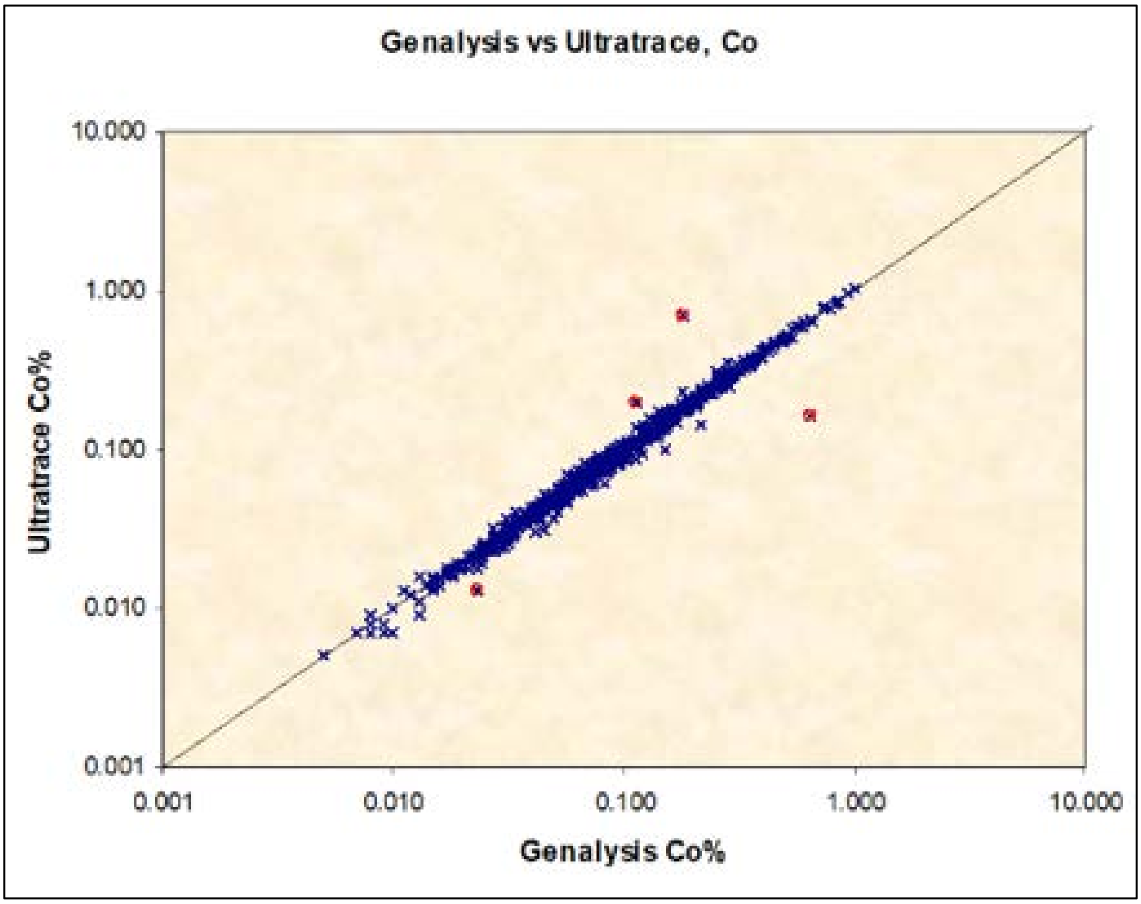

Figure 11-10: Check Assay Scatterplots Genalysis vs Ultra Trace, Cobalt

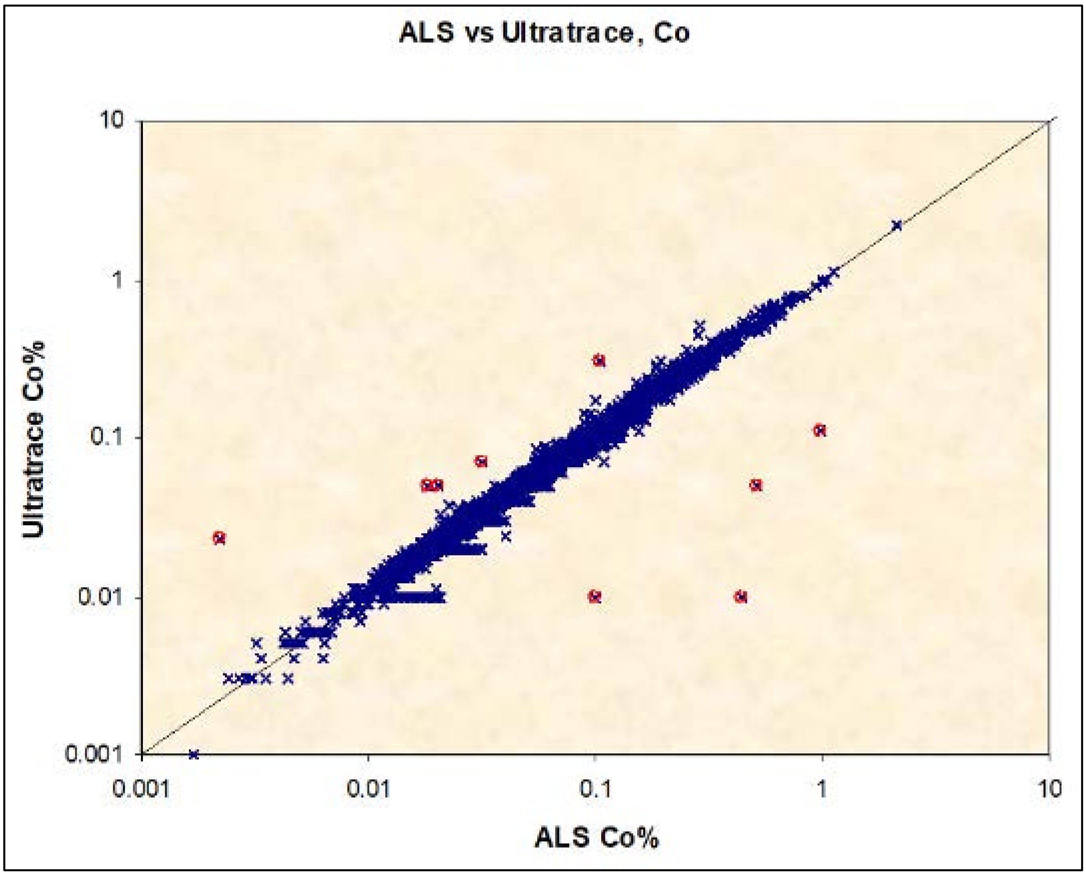

Figure 11-11: Check Assay Scatterplots ALS vs Ultra Trace, Cobalt

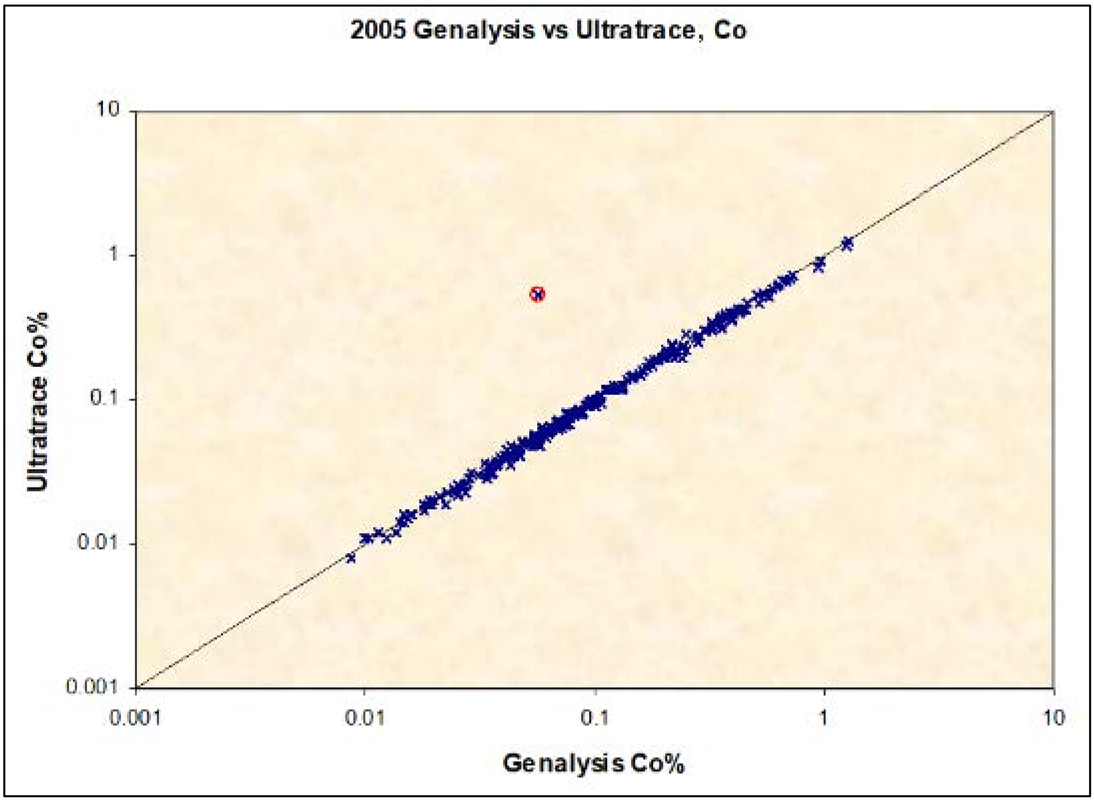

Figure 11-12: Check Assay Scatterplots 2005 Genalysis vs Ultra Trace, Cobalt

11.6.3 Independent custody samples

11.6.3.1 SNC-Lavalin Australia Pty Ltd (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 Ultra Trace values (see Table 11-10), 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 chromium values correlated poorly and there was potential under-reporting of chromium grades by Ultra Trace. This was consistent with other check assay results.

Ultra Trace tended to give lower aluminium values above about 0.8%.

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

Table 11-10: Summary Statistics of SLA Custody Sampling for Nickel and Cobalt (Source: SLA, July 2000)

Statistic	Nickel (ppm)		Cobalt (ppm)	
	Genalysis (SLA)	Ultra Trace (Black Range)	Genalysis (SLA)	Ultra Trace (Black Range)
Count	204	204	204	204
Minimum	145	120	6	10
Maximum	17,000	17,000	5,600	5,650
Mean	7,113	6,989	843	830
Standard deviation	4,661	4,642	1,039	1,039
Correlation coefficient	0.9966		0.9971	

11.6.3.2 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 1m independent sample results for these intervals were consequently averaged and compared with the original 4m composite sample results. In one case, the original samples from the first 4m 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 11-11.

Table 11-11: Summary Statistics of 2005 Independent Custody Samples

Element		All Data				Probable Outliers Excluded				
		Ultra Trace	Genalysis	Mean	Relative Difference	Proportion Excluded	Ultra Trace	Genalysis	Mean	Relative Difference
Nickel (%)	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%
Cobalt (%)	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 nickel compared satisfactorily (see Figure 11-13). Ultra Trace tended to be slightly higher, with a mean relative difference of only about 2%.

Results for cobalt were also satisfactory (see Figure 11-13). Ultra Trace 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 like that indicated by the batches of normal check assays.

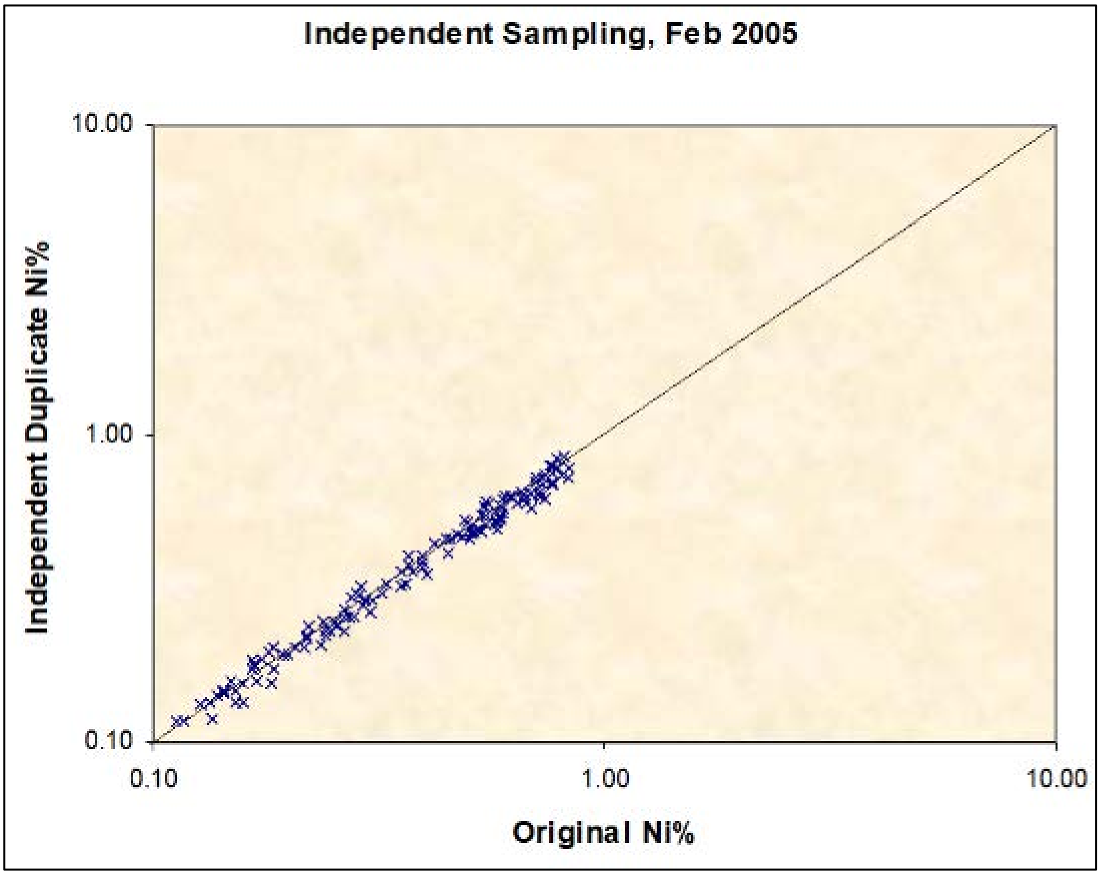

Figure 11-13: Independent Custody Sample Scatterplots for Nickel

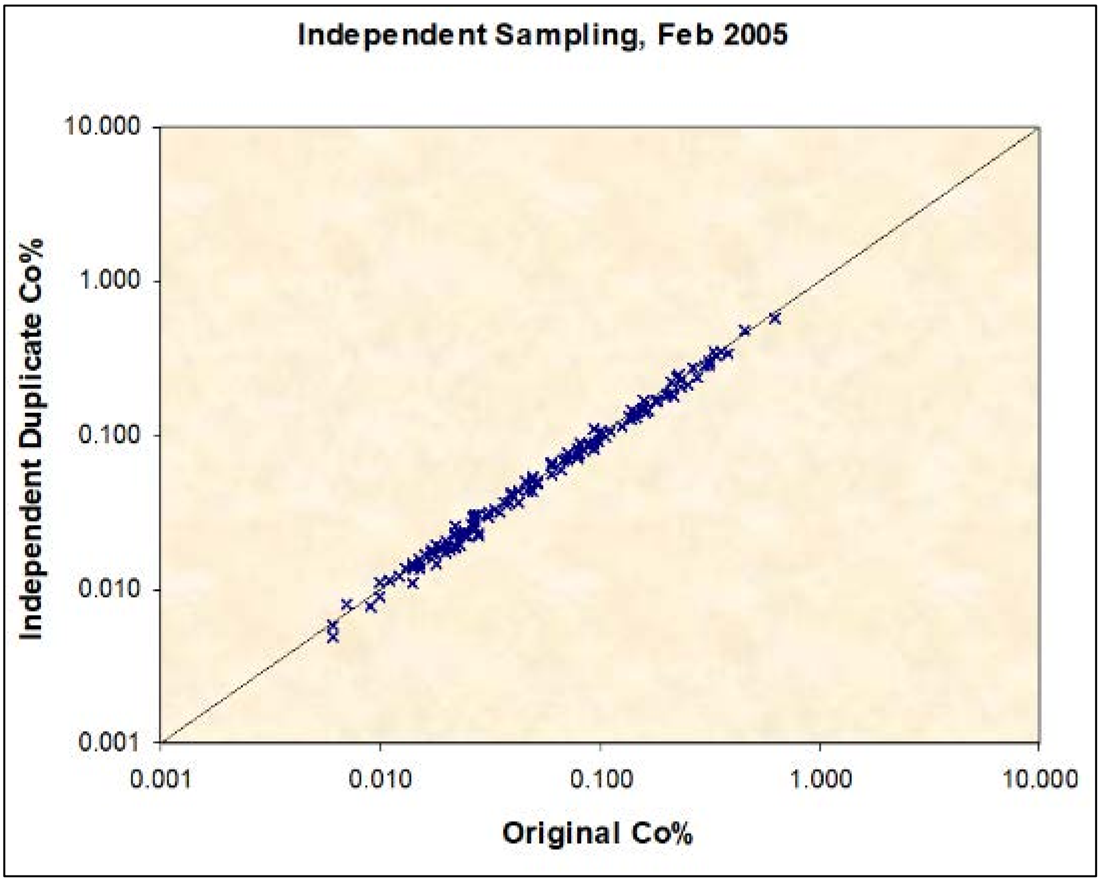

Figure 11-14: Independent Custody Sample Scatterplots for Cobalt

11.6.3.3 Other elements

Results for iron, copper, magnesium, scandium and zinc compared satisfactorily (mean relative differences within about ± 3%).

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

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

Ultra Trace results for calcium also tended to be higher, with a mean relative difference of about 8%.

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

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

11.6.4 Standards

When dealing with standard samples, McDonald Speijers 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.

11.6.5 1999-2000 Results for Standards

McDonald Speijers extracted 766 sets of Ultra Trace 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 differentiate between the different standards based on reasonably distinctive combinations of assay values for various elements, as shown in Figure 11-15 below, e.g. SYS2, 3 and 5 could be distinguished based on substantially different aluminium and cobalt values.

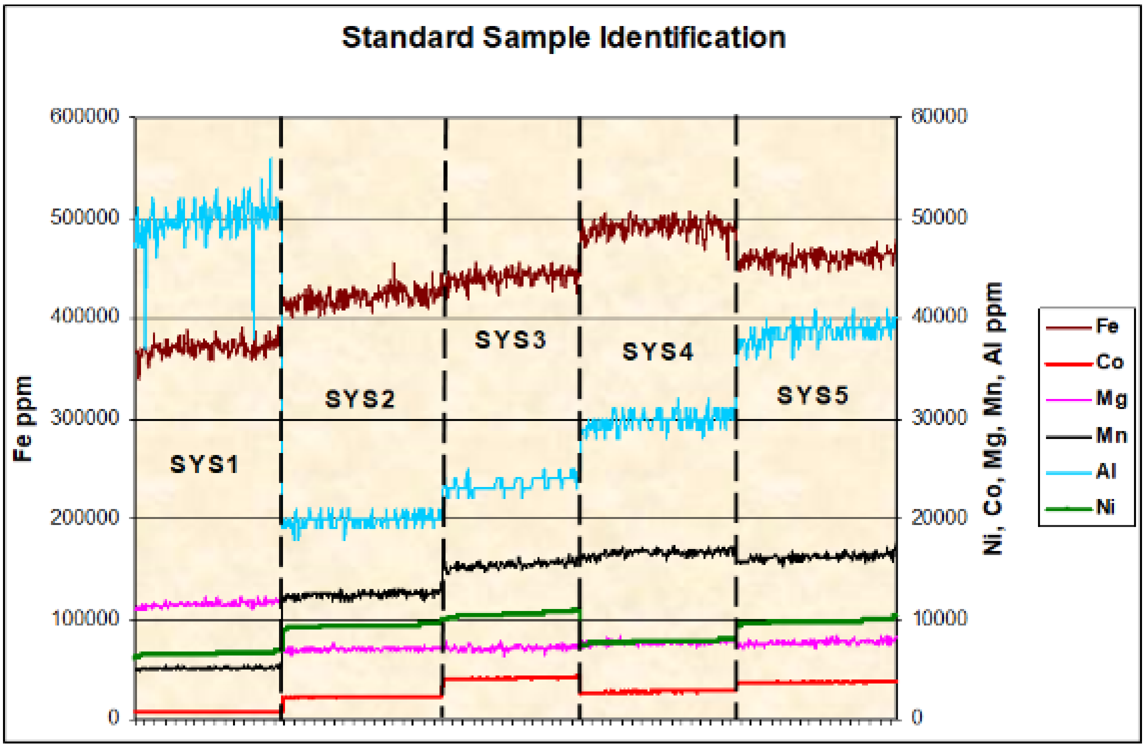

Figure 11-15: Chart Showing Assumed Standard Sample Identifications 1999-2000

There was a general tendency for Ultra Trace 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 11-12: Standard Samples 1999-2000 Summary Statistics

Assumed Standard Number		Nickel (%)	Expected Nickel (%)	Relative Difference (Nickel)	Cobalt (%)	Expected Cobalt (%)
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	-
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 Ultra Trace 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 nickel or cobalt (Table 11-12 above).

11.6.5.1 2005 results for standards

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 Clean TeQ Sunrise 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 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 SD:

- 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 (see Table 11-13).

11.6.5.2 Discussion

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

For the old Clean TeQ Sunrise 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 11-16 and Figure 11-17. 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%. Many of these were outside an acceptable range.

Table 11-13: Standards Results

Standard Number		Ultra Trace Results						Genalysis Results					
		Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt	Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt)
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%

Standard Number		Ultra Trace Results						Genalysis Results					
		Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt	Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt)
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%
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 Number		Ultra Trace Results						Genalysis Results					
		Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt	Nickel (%)	Expected Nickel (%)	Relative Difference Nickel	Cobalt (%)	Expected Cobalt (%)	Relative Difference Cobalt)
	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%

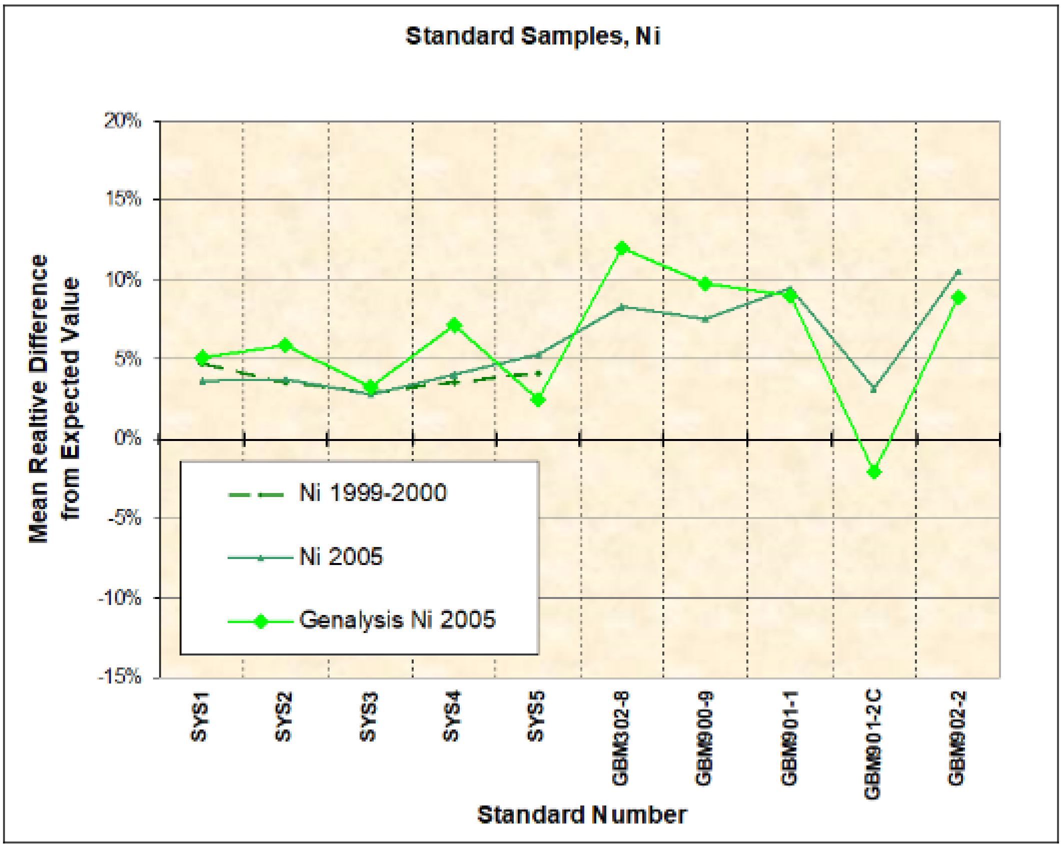

Figure 11-16: Mean Relative Differences from Expected Values of Standards (Nickel)

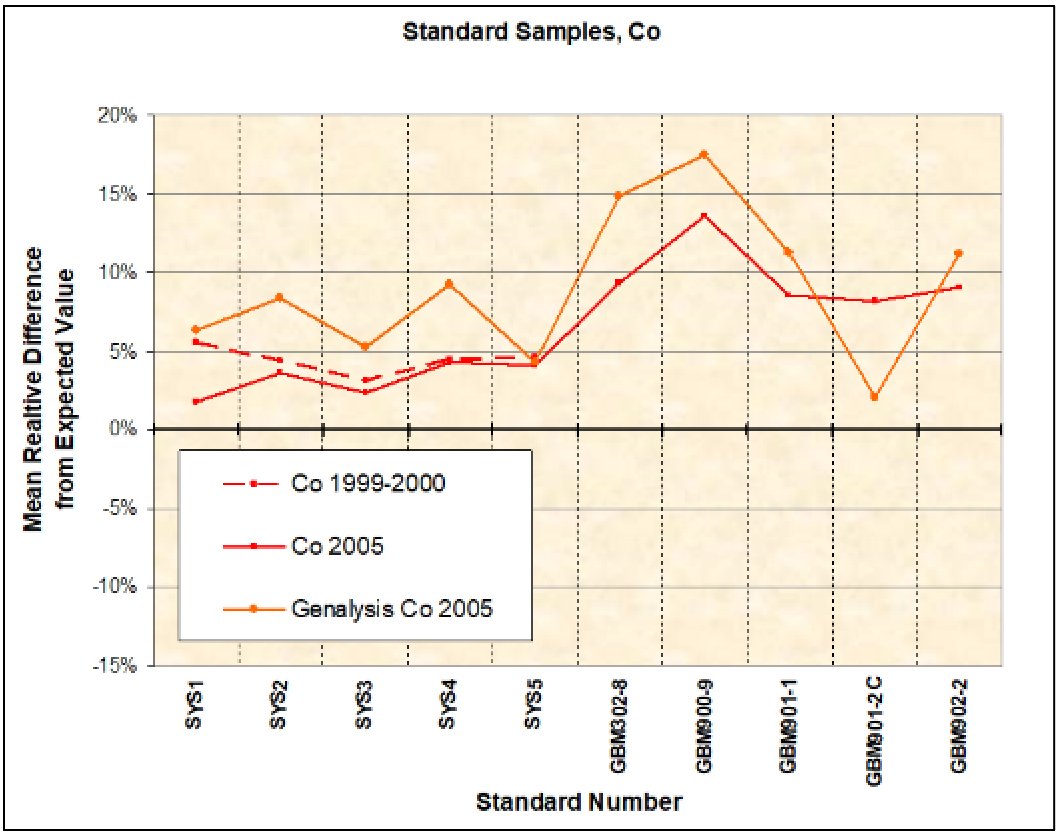

Figure 11-17: Mean Relative Differences from Expected Values of Standards (Cobalt)

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. Additional checks were then conducted to investigate this further.

Ultra Trace 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 (see Table 11-14).

Ultra Trace 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 11-14: Summary Statistics of Ultra Trace Re-Assays of Standard Samples 2005

Statistic	Assays			Relative Differences		
	Original ICP	Repeat ICP	XRF	Repeat Original	XRF Original	XRF Repeat
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 Ultra Trace and Genalysis). The suppliers also reviewed the performance since 2000 of Ultra Trace and Genalysis on standards included in their large, regular round robins. This included comparing these laboratories with available neutron activation analyses for cobalt by Becquerel Laboratories. They reported as follows:

- while individual nickel and cobalt assay results from both laboratories were consistently within acceptable limits, the Ultra Trace 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 (see Table 11-15). This was reasonably like the average 2005 performance on standards that were included with the Clean TeQ Sunrise 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 Ultra Trace remained high.

Table 11-15: Summary Results of Standards Geostats Pty Ltd October 2000 - October 2004

Standard	Expected Value	Standard Deviation	Average Assay				Relative Standardised Difference		
			Preferred Labs	Genalysis	Ultra Trace	Becquerel	Genalysis	Ultra Trace	Becquerel
Nickel									
GBM302-8	10,775	668	10,919	11,300	11,600	-	4.9%	7.7%	-
GBM900-9	11,615	744	11,651	11,400	12,300	-	-1.9%	5.9%	-
GBM901-1	8,037	489	8,120	8,610	8,860	-	7.1%	10.2%	-
GBM901-2	8,804	581	9,057	8,650	8,830	-	-1.8%	0.3%	-
GBM902-2	3,014	285	3,028	3,190	3,270	-	5.8%	8.5%	-
Mean	8,449	553	8,555	8,630	8,972	-	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	1,346	77	1,343	1,279	1,460	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	1,050	1,180	-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, Ultra Trace 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 Ultra Trace 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. Ultra Trace and Genalysis have good reputations for, and are highly experienced in, the analysis of lateritic nickel and cobalt mineralisation and nickel and platinum group element mineralisation in general.

The average result from limited neutron activation analysis (cobalt only) tended to support the Ultra Trace 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 was aware that both Ultra Trace and Genalysis pay a good deal of attention to their acid digest procedures and McDonald Speijers suspected that the apparent bias shown by the standards is likely to reflect more effective digestion of samples than is achieved by most other laboratories.

11.6.5.3 2015 results for standards

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 scandium standard.

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

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 2017 Mineral Resource Estimate QA/QC Drill Hole Series Validation

11.7.1 2017 Assay Result Validation

During the creation of the Clean TeQ Geobank™ CLQGB_MODEL drill hole database the drill hole location data was validated, but when the assay data from the Black Range Minerals Microsoft Access™ Syerston database was investigated there were serious concerns that the assay results from the various laboratories (Ultra Trace, Genalysis and ALS) may have been stored incorrectly.

To overcome this impasse, the original assay results files in either csv or sif format was requested from the participating laboratories. This proved partially successful, however in the case of Ultra Trace (the principal laboratory used), all efforts to recover the original assay results files were unsuccessful.

A major effort was then focussed on identifying these assay records from the archive files and ultimately approximately 70% of the historic assay values were recovered, which included the important metadata required for importation into the Geobank™ database.

Historically, the original assay results files were imported into Micromine™ assay interval files, using either a sample identification or the hole number together with the from and to values, which were concatenated to create a unique identifiable interval reference. The results from the individual laboratories were initially imported and finally combined to form a single assay interval file that was used for the original resource estimate.

Once all the assay results were identified, they were sorted into the various drilling programs and then the individual assay reports were matched with specific drill holes. The original assay results files were stored separately. In addition, a copy was created' as in some of the assay results files additional metadata were required to enable them to be imported into Geobank™.

For each assay file type (either csv or sif) and for each drilling program, a separate assay file template was created in Geobank™ and the distinct items of missing metadata were added from contemporary laboratory records, file dates, etc.

An Import Object was created for each drilling program, to enable the recording of any item of the metadata that had to be added. This aimed to indicate (in the audit trail) the extent of absence of the original metadata.

The metadata required by the Geobank™ sample tracker includes the following:

- job number
- despatch number
- sample identification
- elements
- units
- method
- results
- date received
- date reported.

Most of either the csv or sif assay result files contained most of the metadata, but they required re-formatting to ensure that the header and data records were parsed into the database table format and the correct field data type was present.

The most difficult component of the assay record importation was the identification of the standards records. Initially there were several site standards prepared by Gannett Holdings Pty Ltd which were re-badged under different naming conventions. These had to be 'mapped' to allow them to be seamlessly imported into the QA/QC table within the database.

The other issue encountered in the importing of the assay results file was the identification of the duplicate samples as they were often repeats of assay intervals without unique identifying numbers. These samples required re-mapping to relate the duplicate sample number to the parent sample number. The re-mapping of the duplicate sample numbers was not always successful and in response to this issue separate load files were created to ensure that the correct duplicate number was matched with the appropriate parent sample number.

11.7.2 Quality Assurance/Quality Control Drill Hole Series Validation

To provide assay confidence for the 2017 Mineral Resource Estimate, several quality assurance/ quality control (QA/QC) 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.

11.7.3 Standards

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

This initial RC drilling program included systematic insertion of 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 number. A linear regression line was plotted on the chart to indicate any discernible trend between successive batches.

In November 2017, the QAQC data for SYS1 to SYS5 was revised, as outlined in Kitto, 2017[22]. Assay data was re-compiled in Micromine™ using historical sample interval data, merged with the assay result files directly from the lab assay reports. In addition, metadata for Gannet Standards SYS1 to SYS5 was obtained directly from Adrian Knowles, the retired proprietor of Gannet Holdings Pty Ltd.

Gannet Standards SYS1 to SYS5 were prepared from material sourced from the Sunrise nickel/cobalt deposit and the reference mean was determined from a round robin assay program whereby the samples were submitted to seven different laboratories and the data analysed.

The first of the revised charts displayed a marked improvement in data performance and it was decided to re-run the entire dataset. The result was that all datasets using Gannet Standards SYS1 to SYS5 were found to be statistically in control for nickel and cobalt analyses. Details are presented in Kitto, 2017.

Revised QAQC charts for SYS5 Ni and SYR5 Co in the SRC001 - SRC340 Drilling Program (August 1997 – August 1998) are shown in Figure 11-18 and Figure 11-19.

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[22] Kitto, P.L., 2017. Revised Shewhart Assay Quality Control Charts Using New Metadata. Clean TeQ Holdings Ltd., Internal Company Memorandum, 1 November 2017 [Unpublished].

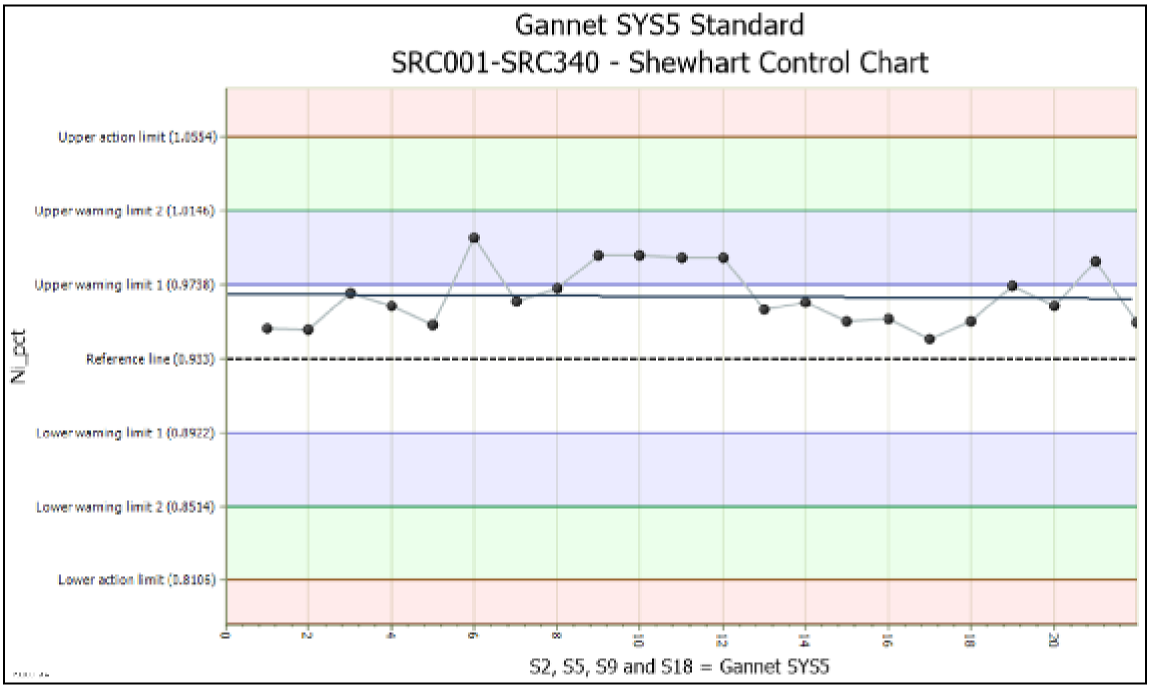


Figure 11-18: Nickel % Shewhart Quality Control Chart - Gannet SYS5 Standard

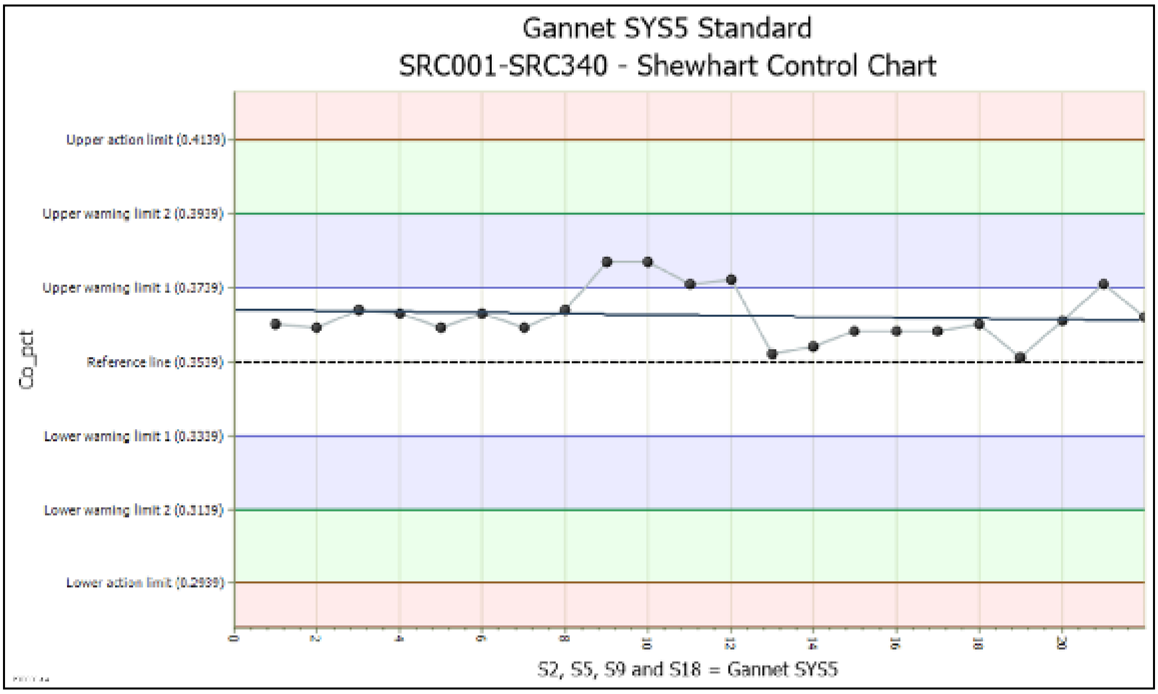


Figure 11-19: Cobalt % Shewhart Quality Control Chart - Gannet SYS5 Standard

11.7.3.2 SRC0341 - SRC1004 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 as illustrated in Figure 11-20 and Figure 11-21 below for Gannet Standard SYS4 (nickel and cobalt).

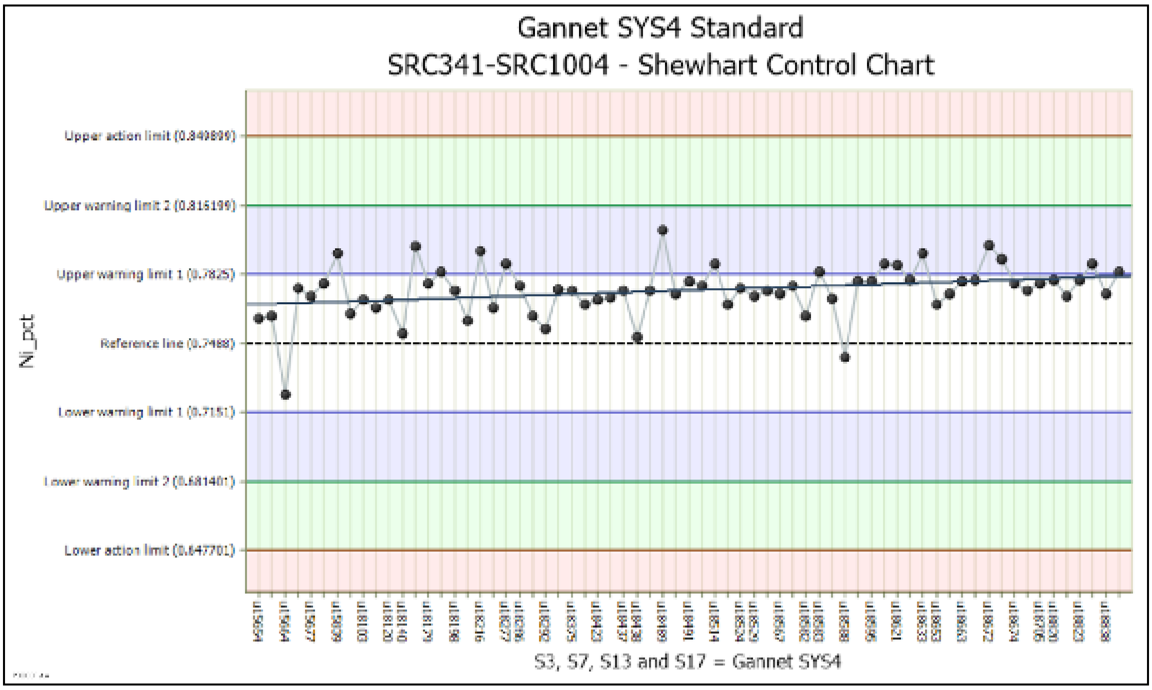


Figure 11-20: Nickel % Shewhart Quality Control Chart - Gannet SYS4 Standard

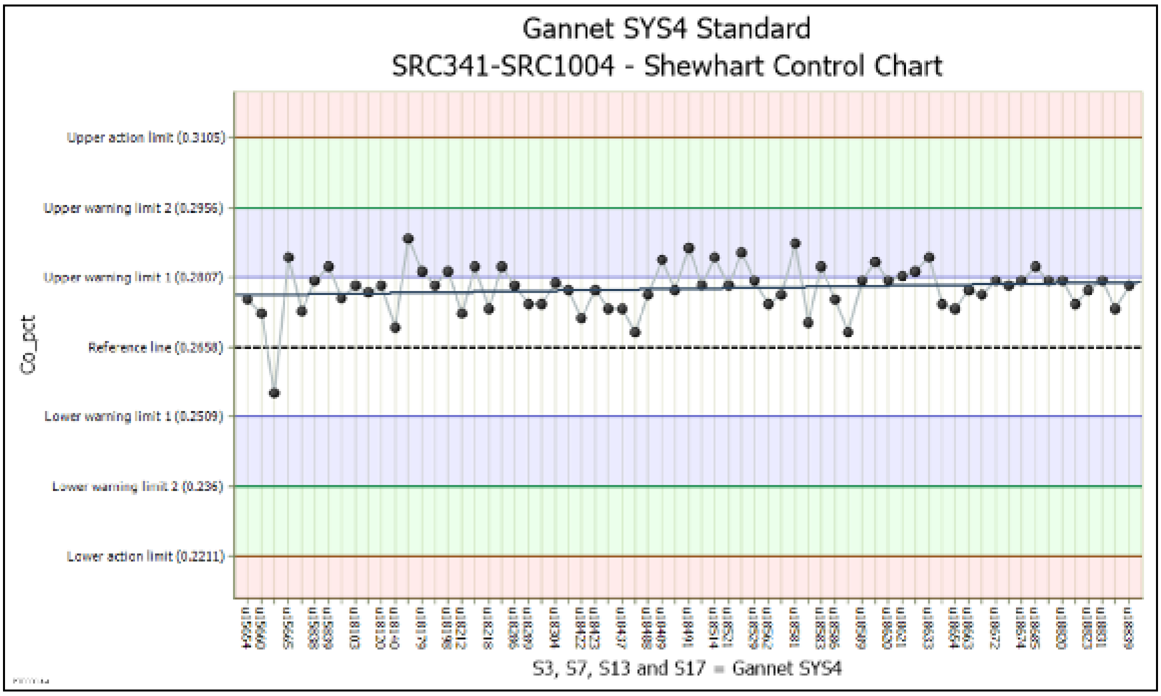


Figure 11-21: Cobalt % Shewhart Quality Control Chart - Gannet SYS4 Standard

All values for the SRC341 – SRC1004 programme were in control.

11.7.3.3 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.

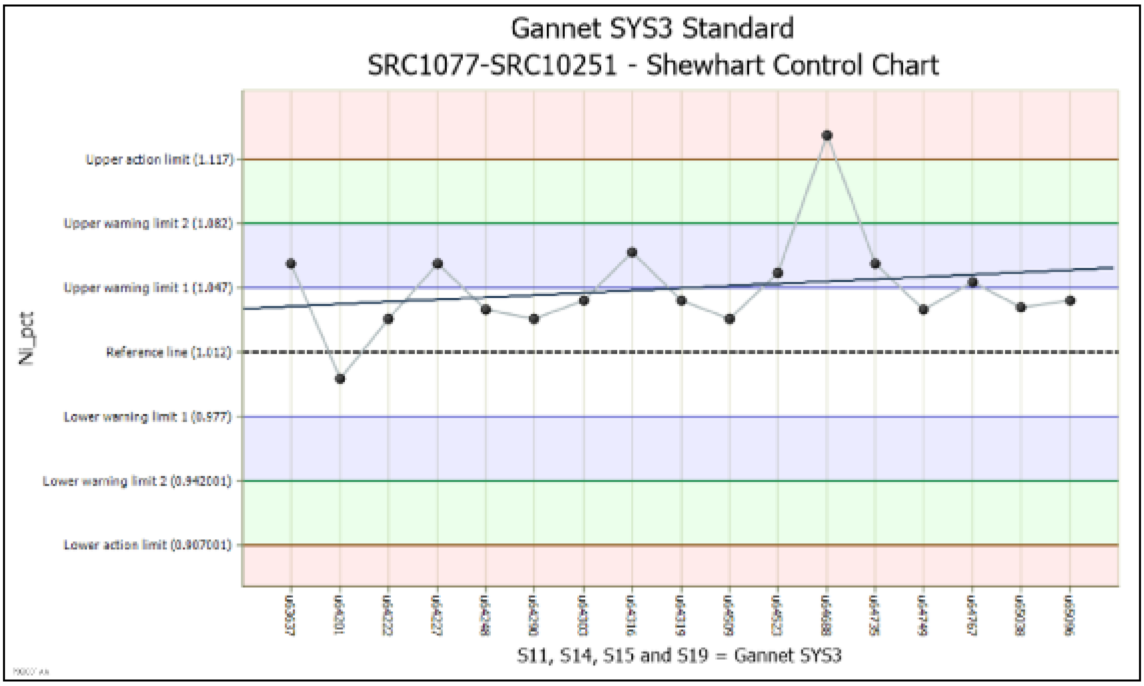


Figure 11-22: Nickel % Shewhart Quality Control Chart - Gannet SYS3 Standard

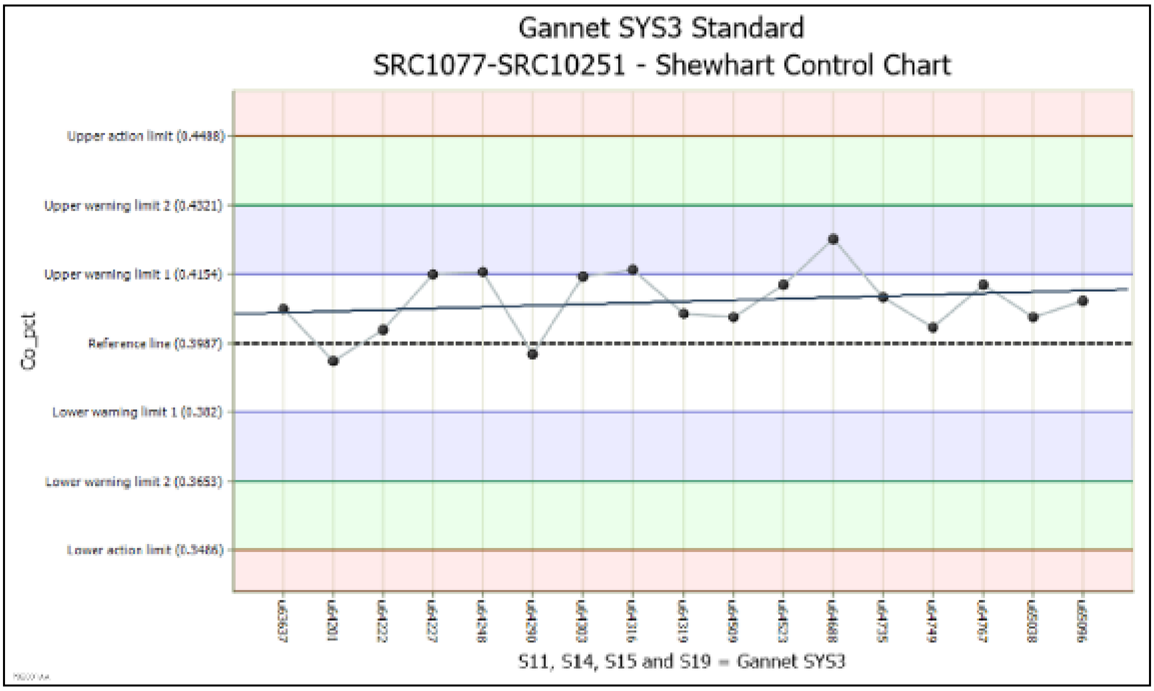


Figure 11-23: Cobalt % Shewhart Quality Control Chart - Gannet SYS3 Standard

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 2 standard deviations (SDs) and the number of outliers (+3 SDs) is very minor (see Figure 11-22 and Figure 11-23 above). The batches are statistically reliable.

11.7.3.4 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.

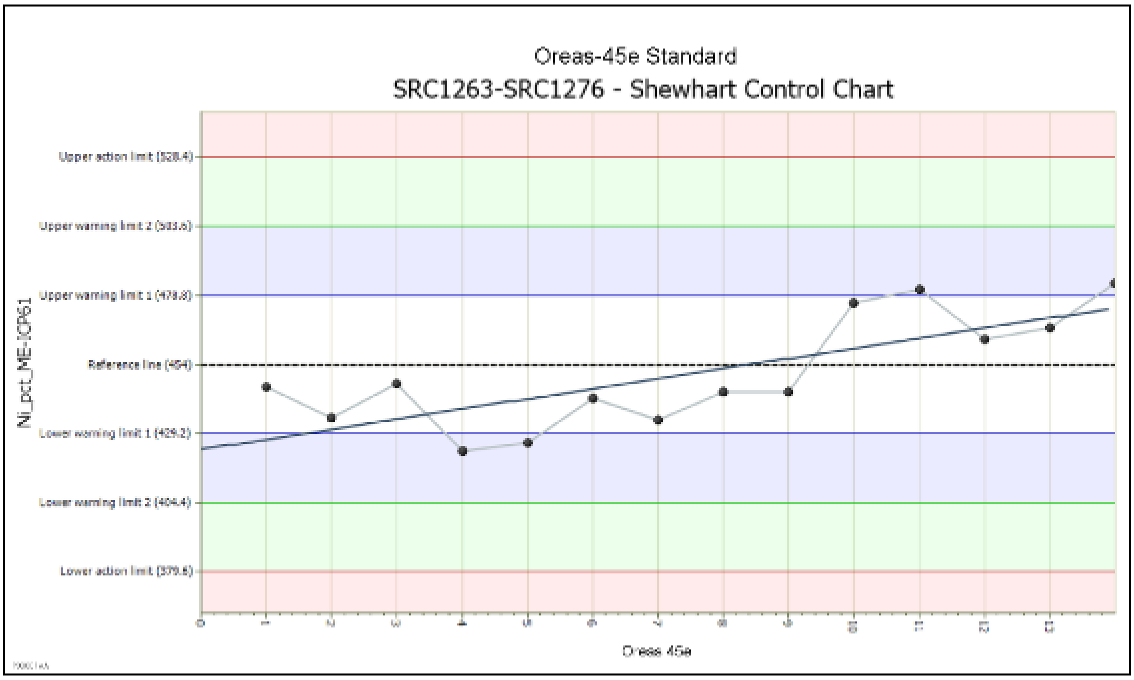


Figure 11-24: Nickel % Shewhart Quality Control Chart - OREAS 45e Standard

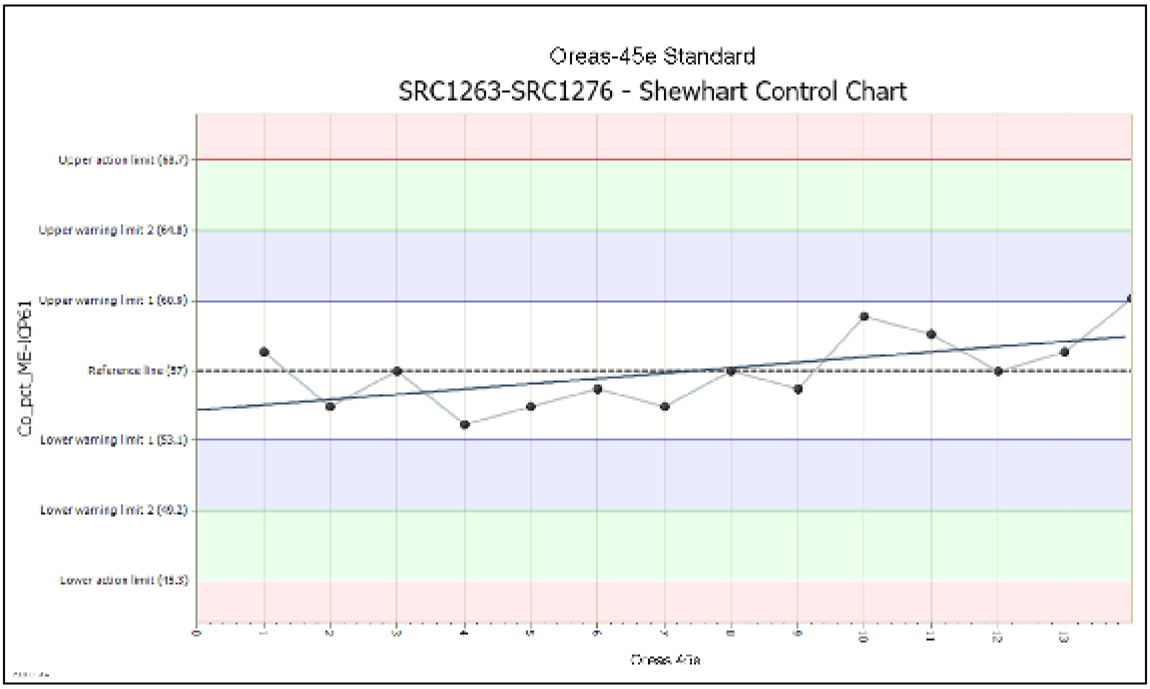


Figure 11-25: Cobalt % Shewhart Quality Control Chart - OREAS 45e Standard

The standard results were all within the one laboratory Job No. and although the values for both nickel and cobalt did not exceed ±2 SDs, there is a very discernible trend from the first to last result and it relates both to the nickel and cobalt values (see Figure 11-24 and Figure 11-25 above). This suggests possible drift in the instrument and warranted checking with the laboratory.

11.7.3.5 SRC1277-SRC1310 Drilling Program (April 2015)

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

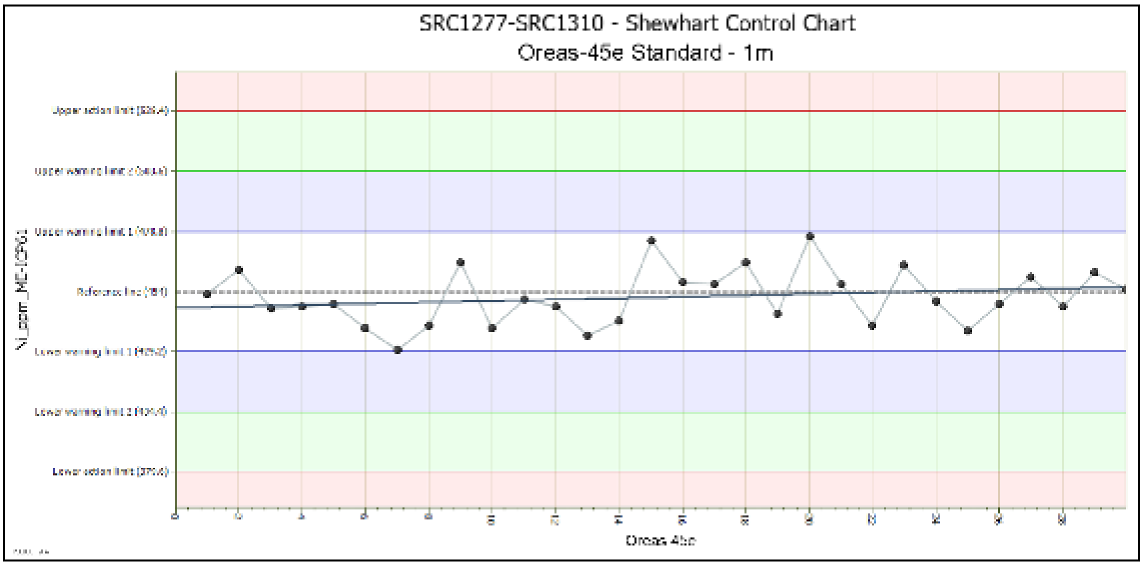


Figure 11-26: Nickel PPM Shewhart Quality Control Chart - OREAS 45e Standard

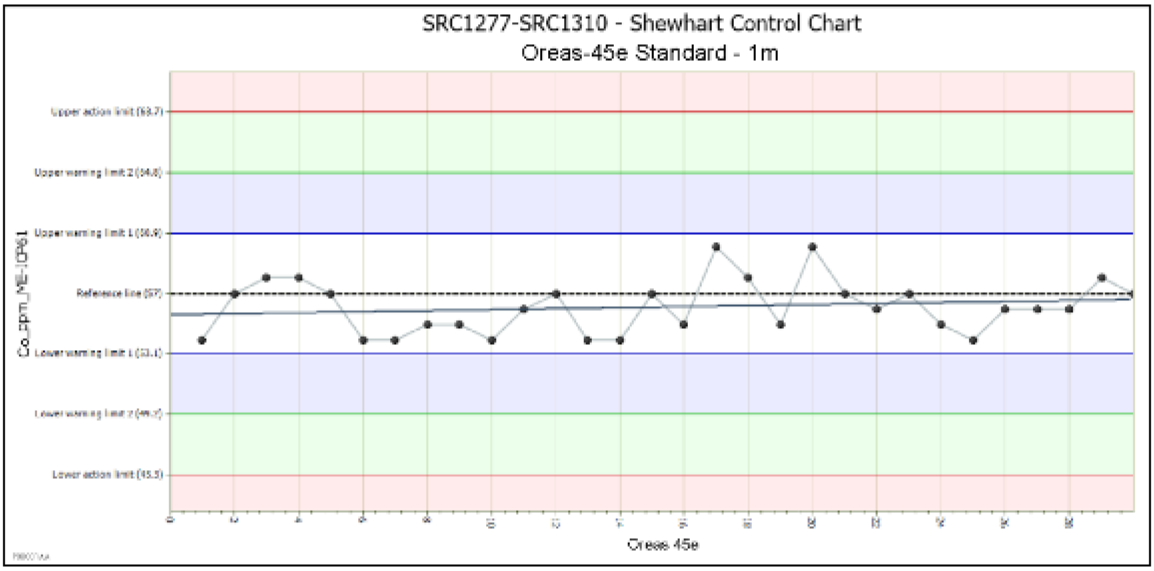


Figure 11-27: Cobalt ppm Shewhart Quality Control Chart - OREAS 45e Standard

The results for nickel improved with the values distributed both sides of the reference line and within ±2 SDs and only a minor trend up in the sequence of values. The cobalt values lie primarily below the reference line with no values exceeding ±2 SDs and a minor up trend in the sequence of values (see Figure 11-26 and Figure 11-27 above).

The lower reported value for cobalt 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 scandium assay quality control chart is included (see Figure 11-28), which indicates that the values are position on the -1 SD line and display an even trend. The slightly lower standard values should not influence the outcome of the routine assay values.

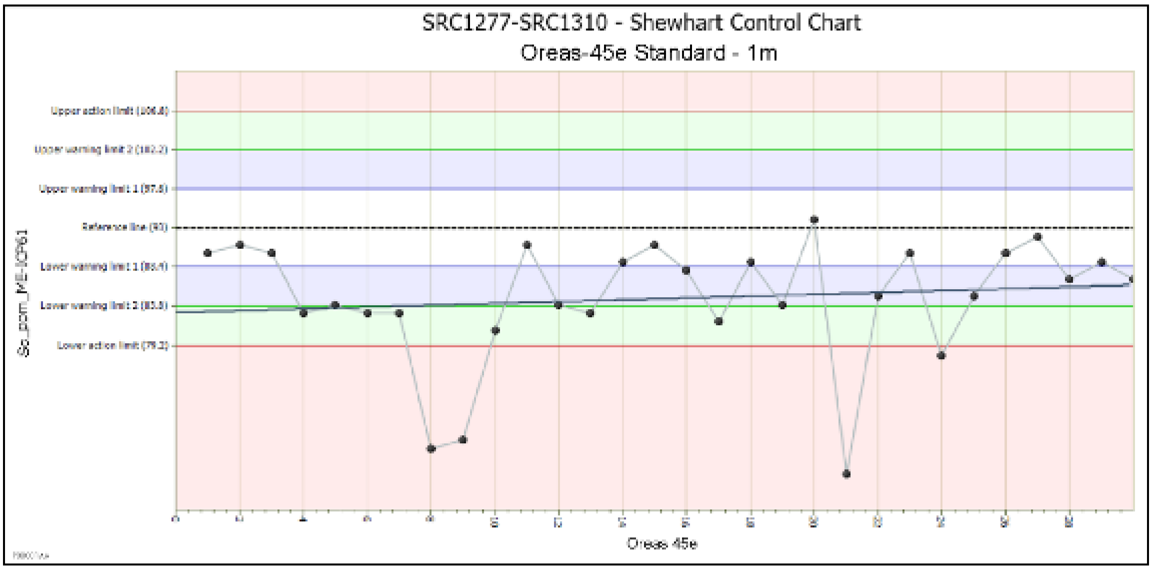


Figure 11-28: Scandium PPM Shewhart Quality Control Chart - OREAS 45e Standard

11.7.3.6 SRC1311-SRC1368 Drilling Program (Nov2015)

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

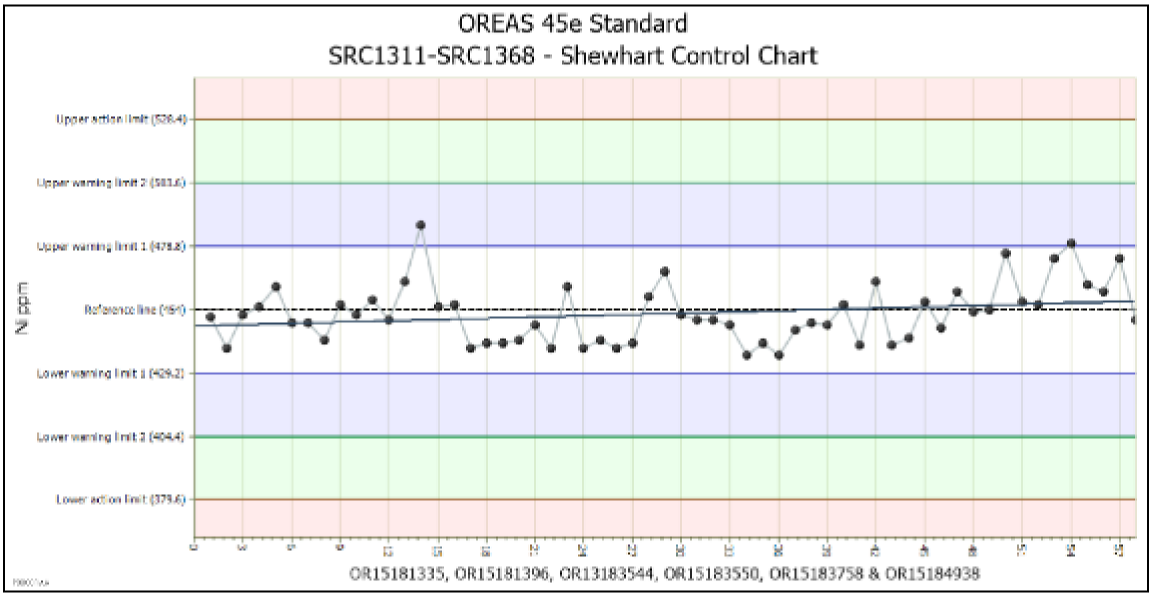


Figure 11-29: Nickel PPM Shewhart Quality Control Chart - OREAS 45e Standard

The nickel standard values straddle the reference line and the values do not exceed ±2 SDs (see Figure 11-29). There is a slight upward trending for the period within which the batches were analysed.

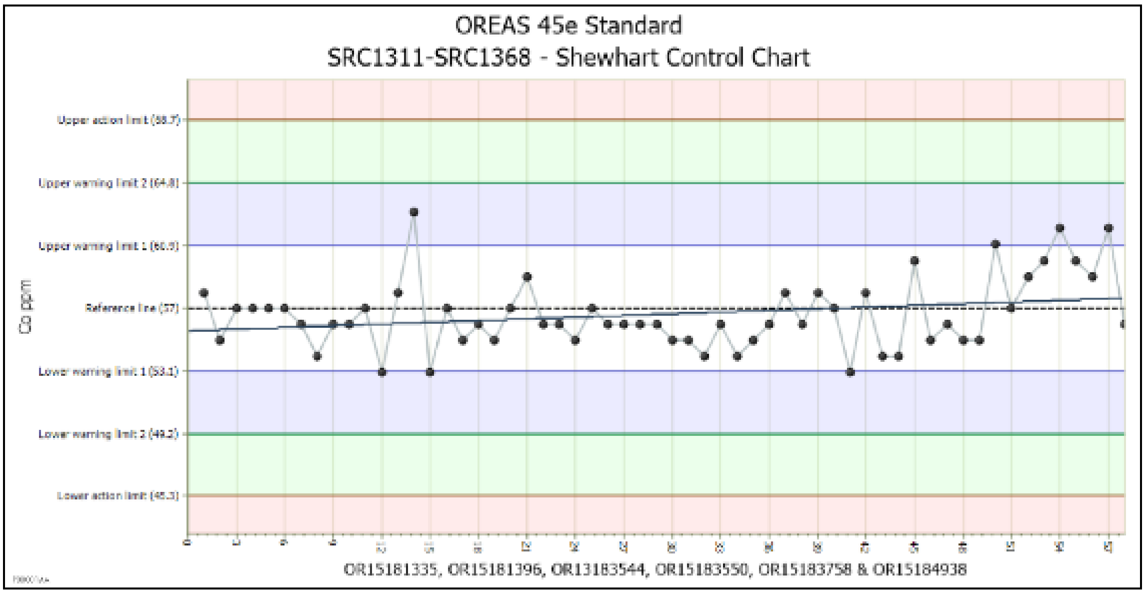


Figure 11-30: Cobalt PPM Shewhart Quality Control Chart - OREAS 45e Standard

The cobalt values are predominantly with -1 SD with a minor number within -2 SDs (see Figure 11-30). 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.

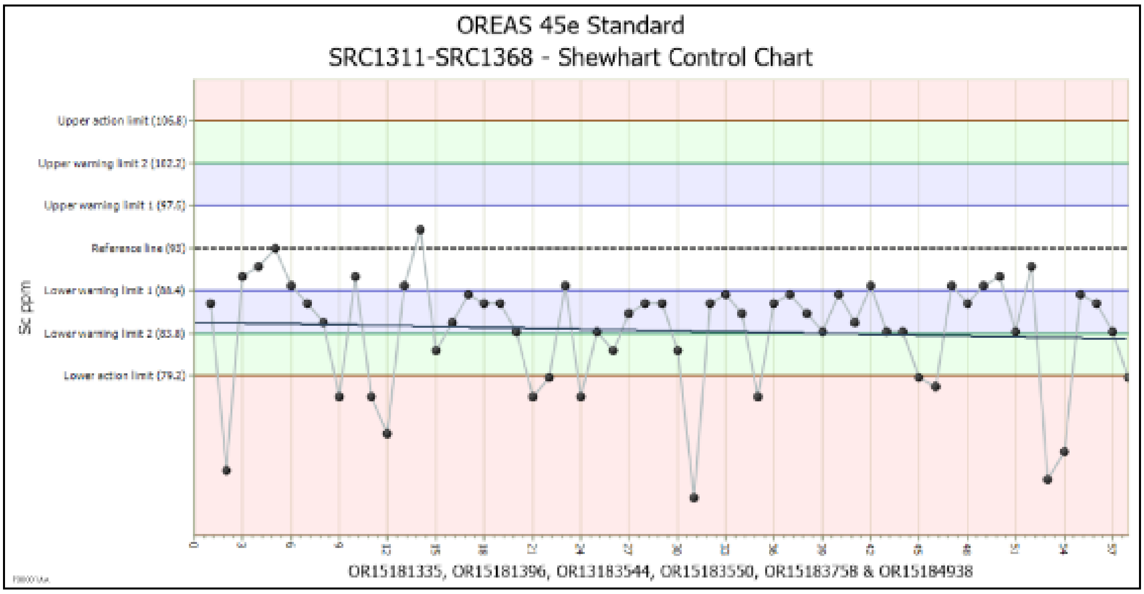


Figure 11-31: Scandium PPM Shewhart Quality Control Chart - OREAS 45e Standard

The scandium values lie either close to or predominantly immediately below the reference line. For the most part, they lie within -1 SD with a minor number -2 SDs and a few outliers (see Figure 11-31).

11.7.4 Duplicates

11.7.4.1 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.

11.7.4.2 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 2018 software. A linear regression is undertaken and the correlation coefficient is determined. A QA/QC limits test is applied at 10%, 20% and 30%.

Nickel duplicate samples (see Figure 11-32):

- correlation coefficient: 0.98
- diagonal verses regression lines: no bias
- ±10% QA/QC limits: 0.5%
- statistical status: statistically in control

Cobalt duplicate samples (see Figure 11-33):

- correlation coefficient: 0.97
- diagonal verses regression lines: no bias
- ±10% QA/QC limits: 2.1%
- statistical status: statistically in control

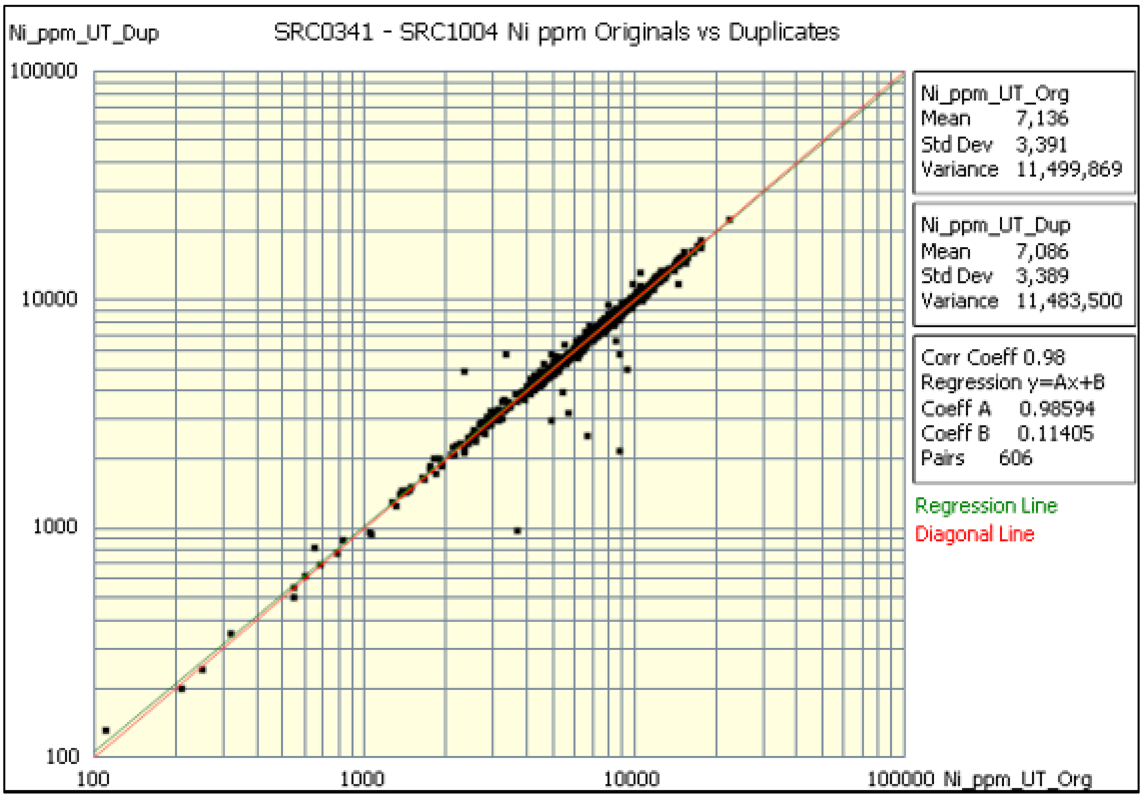


Figure 11-32: SRC0341 - SRC1004 Nickel PPM Originals vs Duplicates

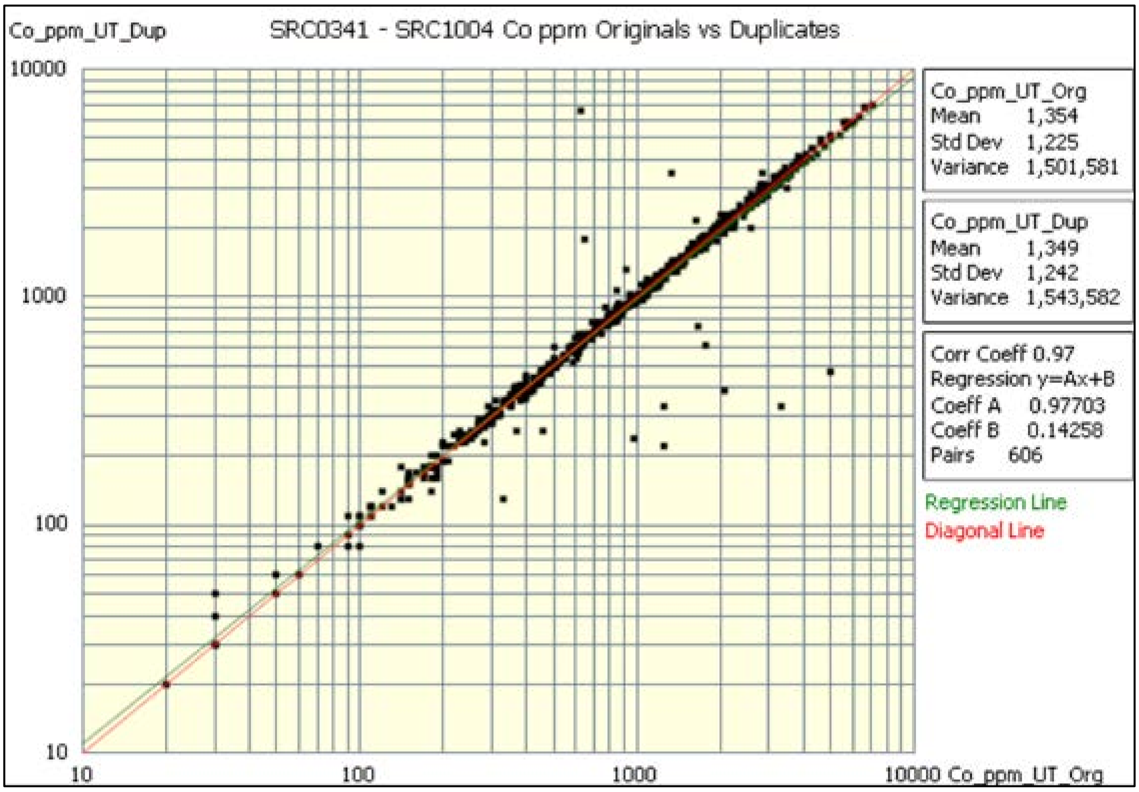


Figure 11-33: SRC0341 - SRC1004 Cobalt PPM Originals vs Duplicates

11.7.4.3 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.

11.7.4.4 SRC1263-SRC1276 Drilling Program (Aug2014)

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

Original verses duplicate tests included:

- correlation coefficient
- diagonal line (1:1) verses regression line (bias test)
- QA/QC limits test at 10%, 20% and 30%.

Nickel duplicate samples (see Figure 11-34):

- correlation coefficient:	1.00
- diagonal verses regression lines:	slight bias towards duplicates
- ±10% QA/QC limits:	0%
- statistical status:	statistically in control

Cobalt duplicate samples (see Figure 11-35):

- correlation coefficient:	0.98
- diagonal verses regression lines:	minor bias towards duplicates
- ±10% QA/QC limits:	15.4%
- statistical status:	statistically out of control

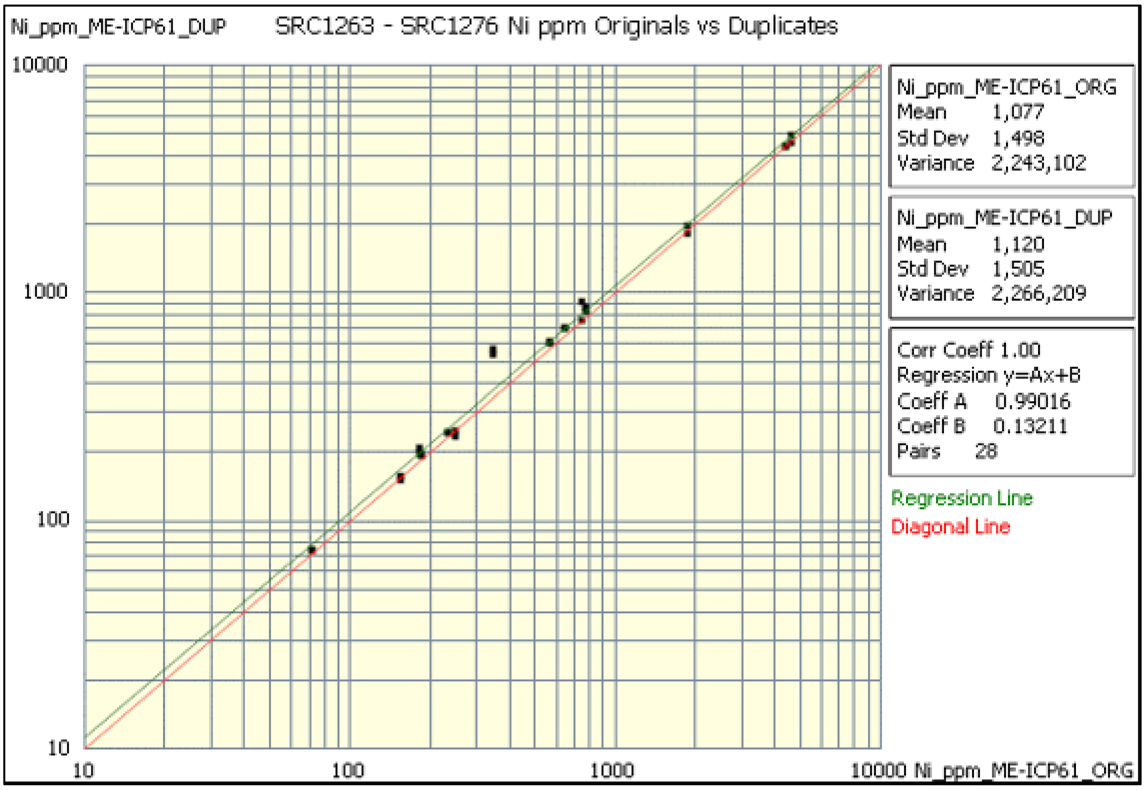


Figure 11-34: SRC1263 - SRC1276 Nickel PPM Originals vs Duplicates

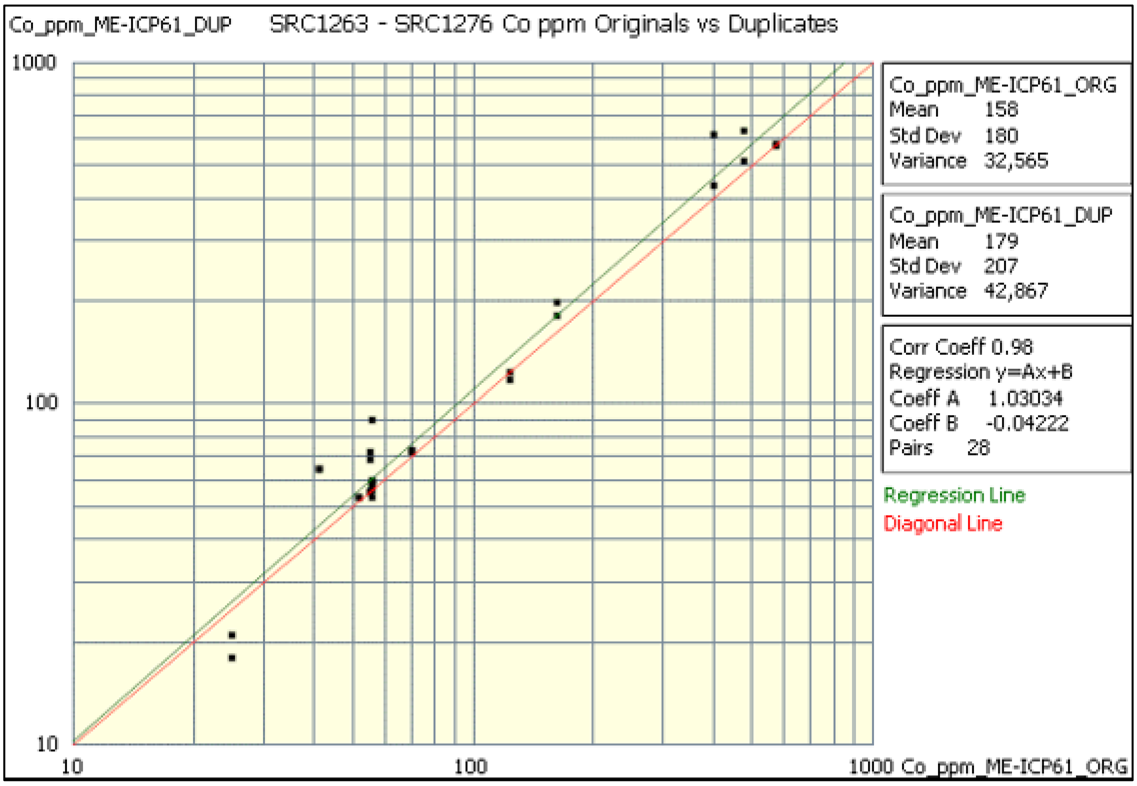


Figure 11-35: SRC1263 - SRC1276 Cobalt PPM Originals vs Duplicates

11.7.4.5 SRC1277-SRC1310 Drilling Program (April 2015)

This 34-hole drilling program was a continuation of the scandium 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 verses duplicate tests included:

- correlation coefficient
- diagonal line (1:1) verses regression line (bias test)
- QA/QC limits test at 10%, 20% and 30%.

2m nickel duplicate samples (see Figure 11-36):

- correlation coefficient: 0.99
- diagonal verses regression lines: pronounced bias towards duplicates at lower values
- ±10% QA/QC limits: 4.4%
- statistical status: statistically in control

2m cobalt duplicate samples (see Figure 11-37):

- correlation coefficient: 0.97
- diagonal verses regression lines: significant bias towards duplicates at lower values
- ±10% QA/QC limits: 19.7%
- statistical status: statistically out of control

2m scandium duplicate samples (see Figure 11-38):

- correlation coefficient: 0.94
- diagonal verses regression lines: slight bias towards originals across the value range
- ±10% QA/QC limits: 16.7%
- statistical status: statistically out of control

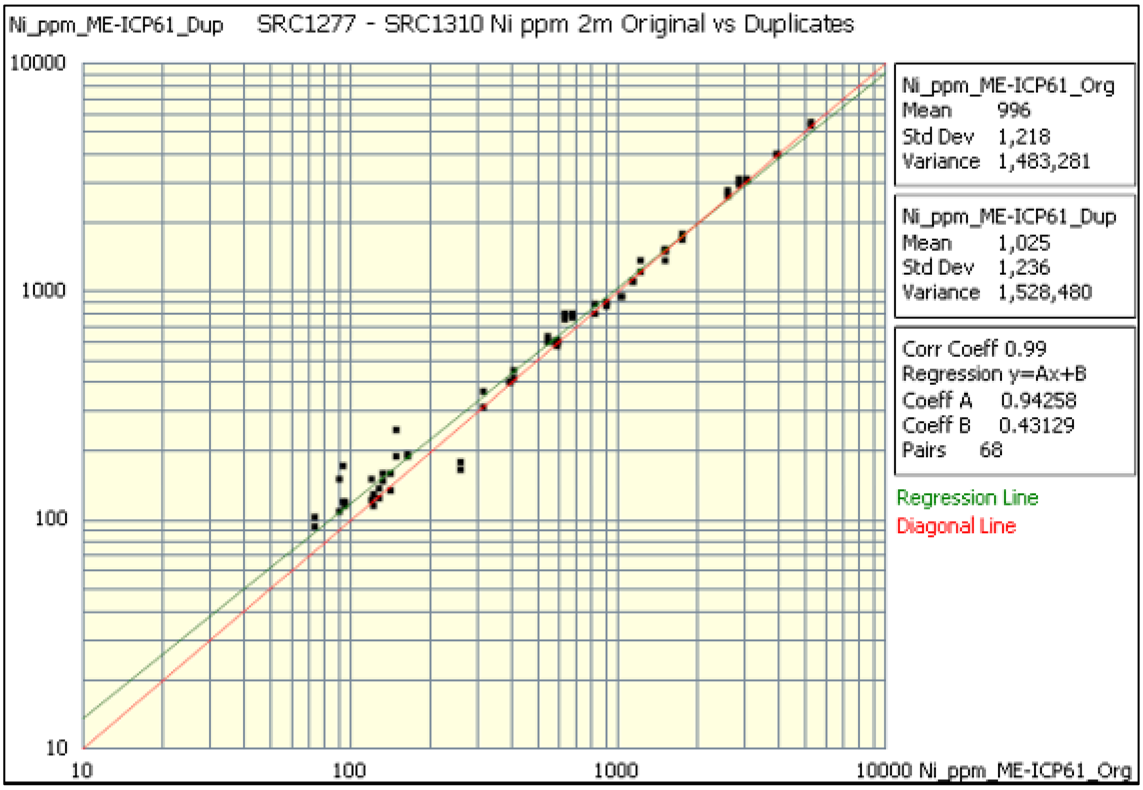


Figure 11-36: SRC1277 - SRC1310 Nickel PPM 2m Originals vs Duplicates

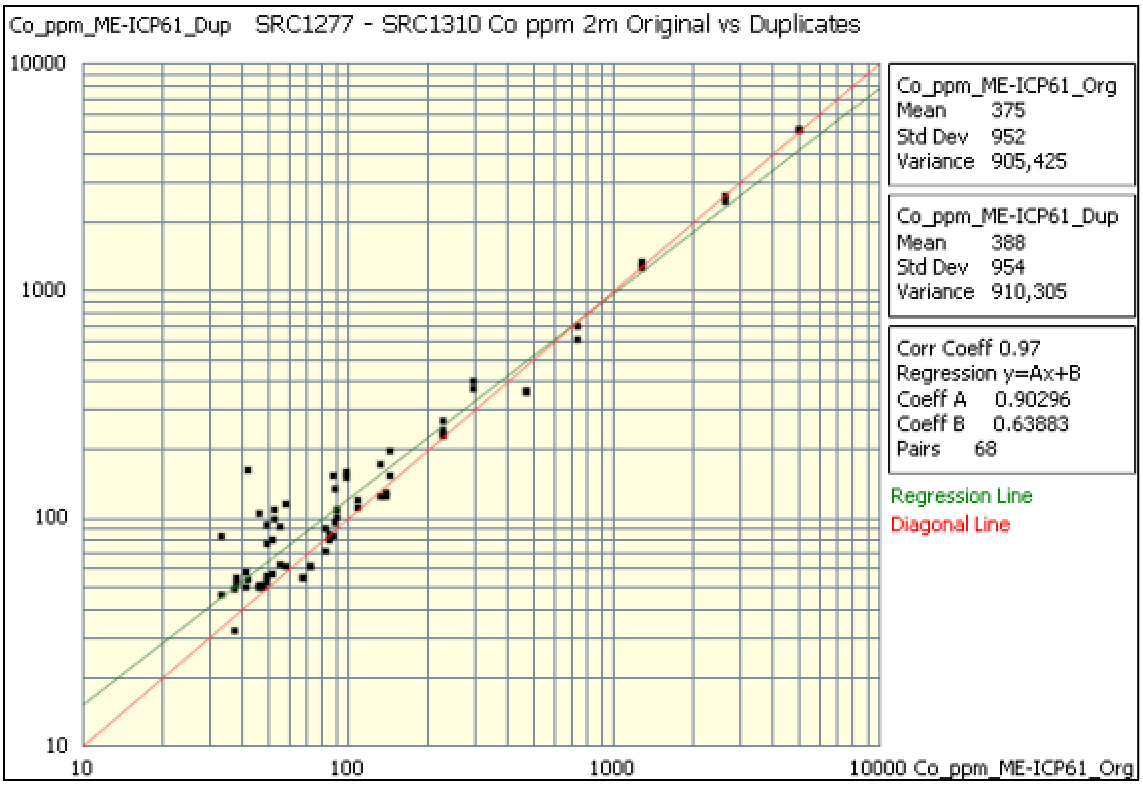


Figure 11-37: SRC1277 - SRC1310 Cobalt PPM 2m Originals vs Duplicates

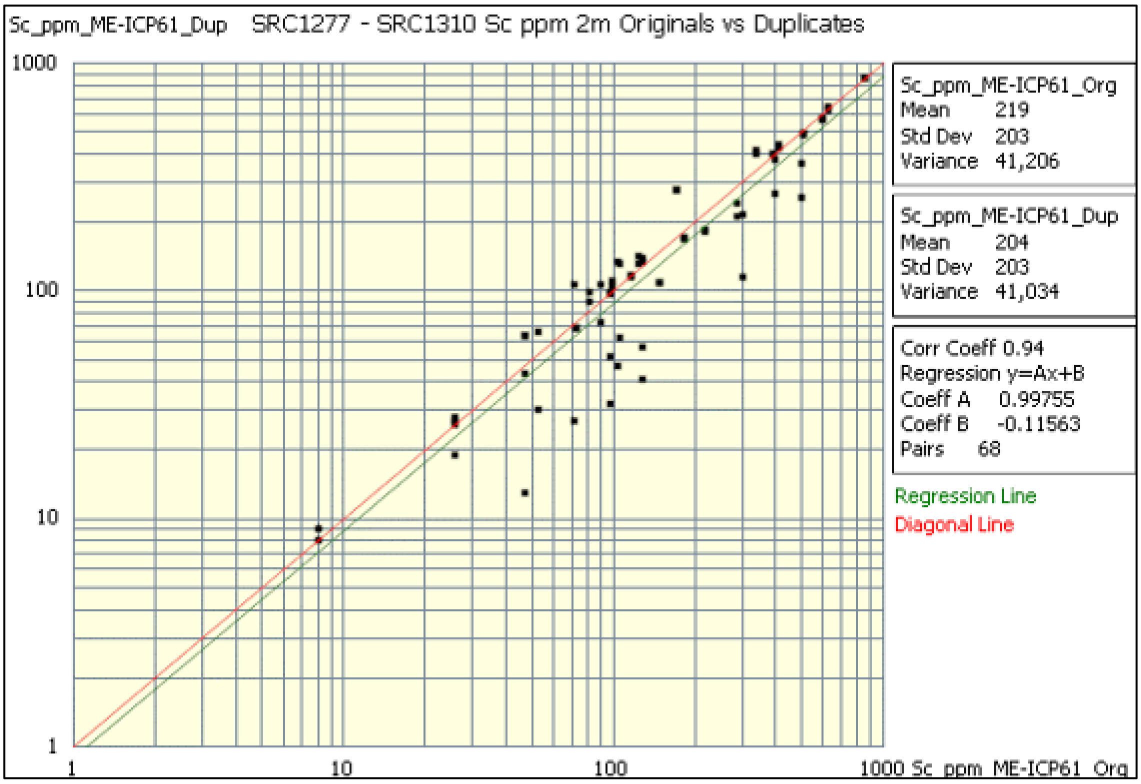


Figure 11-38: SRC1277 - SRC1310 Scandium PPM 2m Original vs Duplicates

1m nickel duplicate samples (see Figure 11-39):

- correlation coefficient:	0.99
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	10%
- statistical status:	statistically in control

1m cobalt duplicate samples (see Figure 11-40):

- correlation coefficient:	0.99
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	0%
- statistical status:	statistically in control

1m scandium duplicate samples (see Figure 11-41):

- correlation coefficient:	0.97
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	0%
- statistical status:	statistically in control

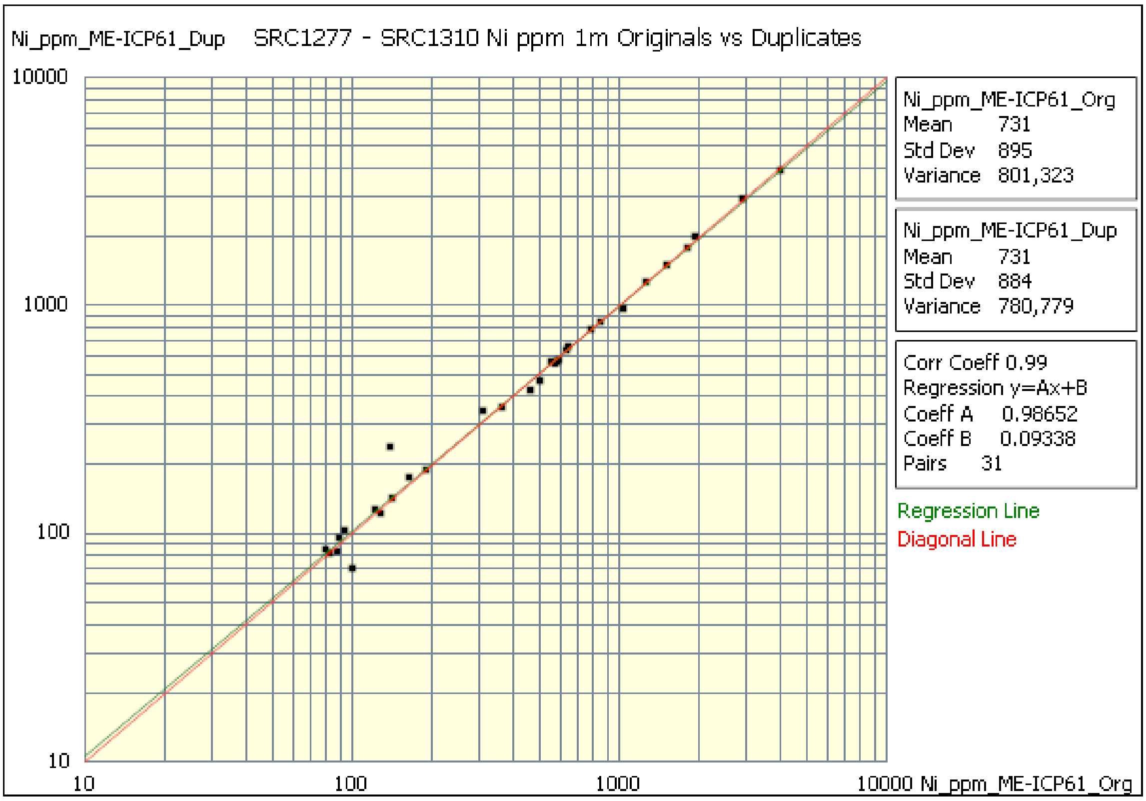


Figure 11-39: SRC1277 - SRC1310 Nickel PPM 1m Originals vs Duplicates

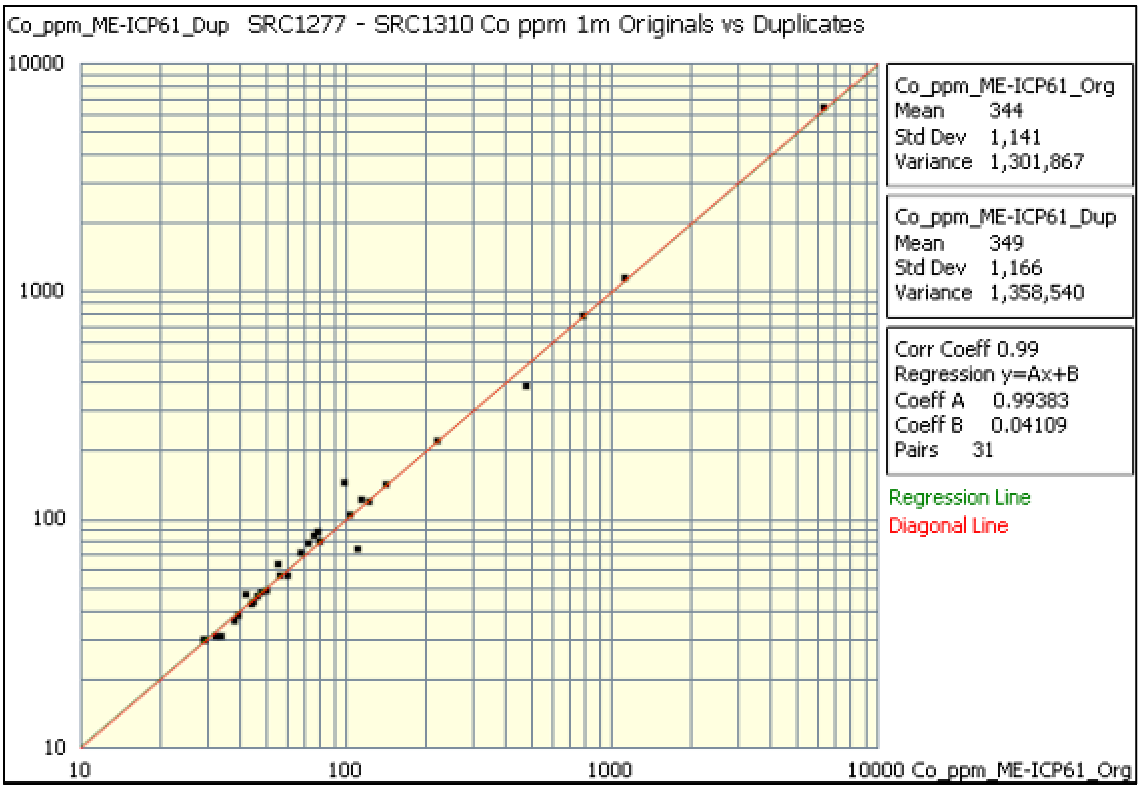


Figure 11-40: SRC1277 - SRC1310 Cobalt PPM 1m Original vs Duplicates

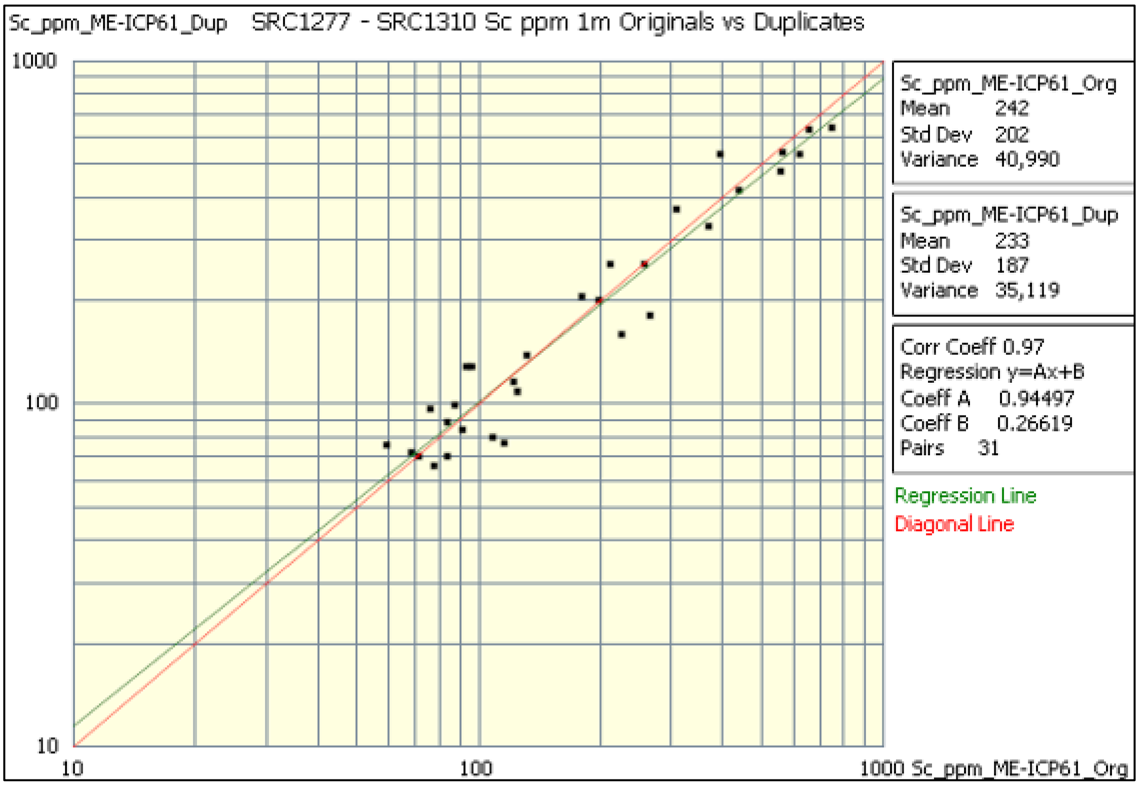


Figure 11-41: SRC1277 - SRC1310 Scandium PPM 1m Originals vs Duplicates

11.7.5 SRC1311-SRC1368 Drilling Program (November 2015)

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

Original verses duplicate tests included:

- correlation coefficient
- diagonal line (1:1) verses regression line (bias test)
- QA/QC limits test at 10%, 20% and 30%.

SRC1311 – SRC1368 nickel duplicate samples (see Figure 11-42):

- correlation coefficient:	0.99
- diagonal verses regression lines:	slight bias to duplicates
- ±10% QA/QC limits:	47.4%
- statistical status:	statistically out of control

SRC1311 – SRC1368 cobalt duplicate samples (see Figure 11-43):

- correlation coefficient:	0.97
- diagonal verses regression lines:	pronounced bias to originals in higher values
- ±10% QA/QC limits:	74.1%
- statistical status:	statistically out of control

SRC1311 – SRC1368 scandium duplicate samples (see Figure 11-44):

- correlation coefficient:	0.98
- diagonal verses regression lines:	no bias

- ±10% QA/QC limits: 51.8%
- statistical status: statistically out of control

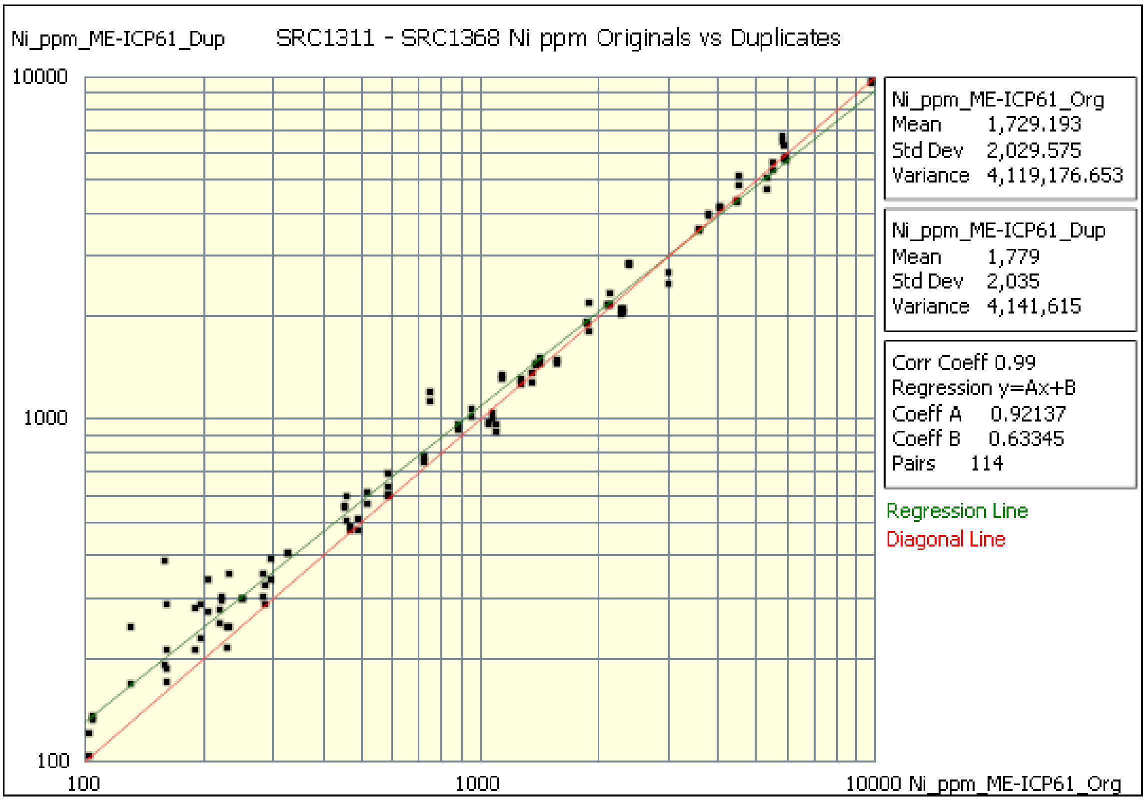


Figure 11-42: SRC1311 - SRC1368 Nickel PPM Originals vs Duplicates

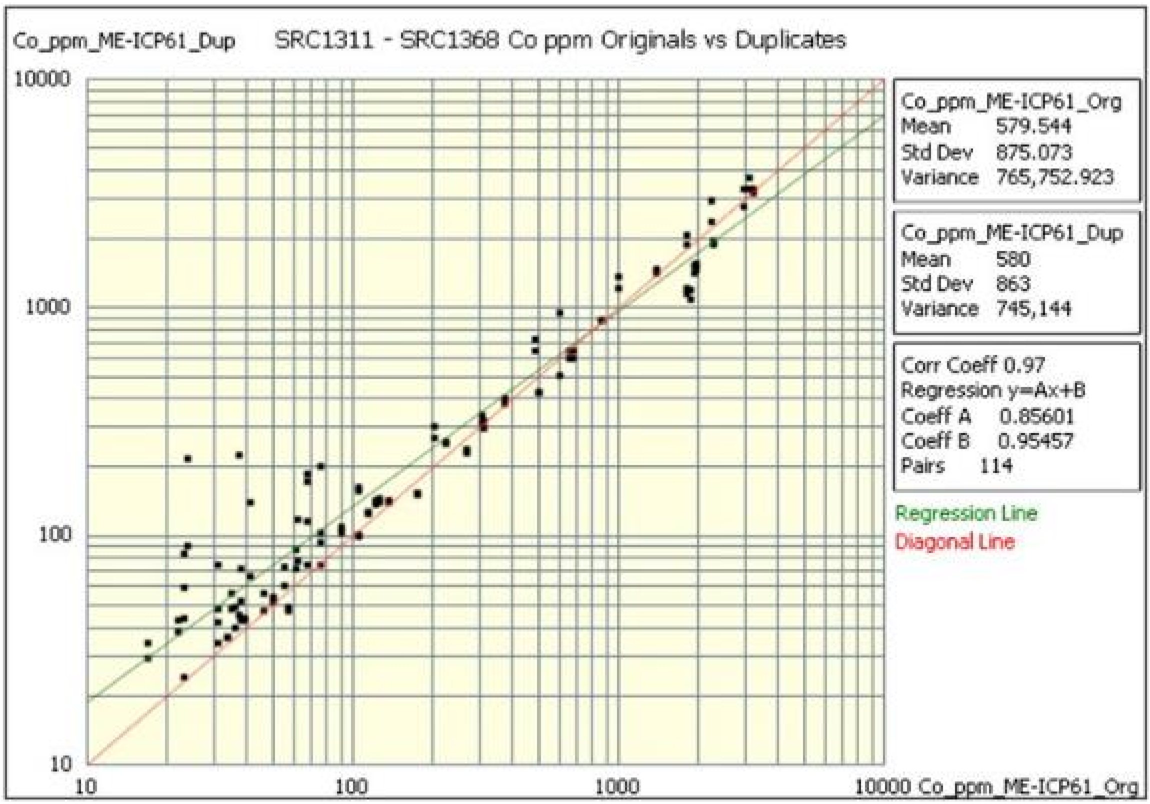


Figure 11-43: SRC1311 - SRC1368 Cobalt PPM Originals vs Duplicates

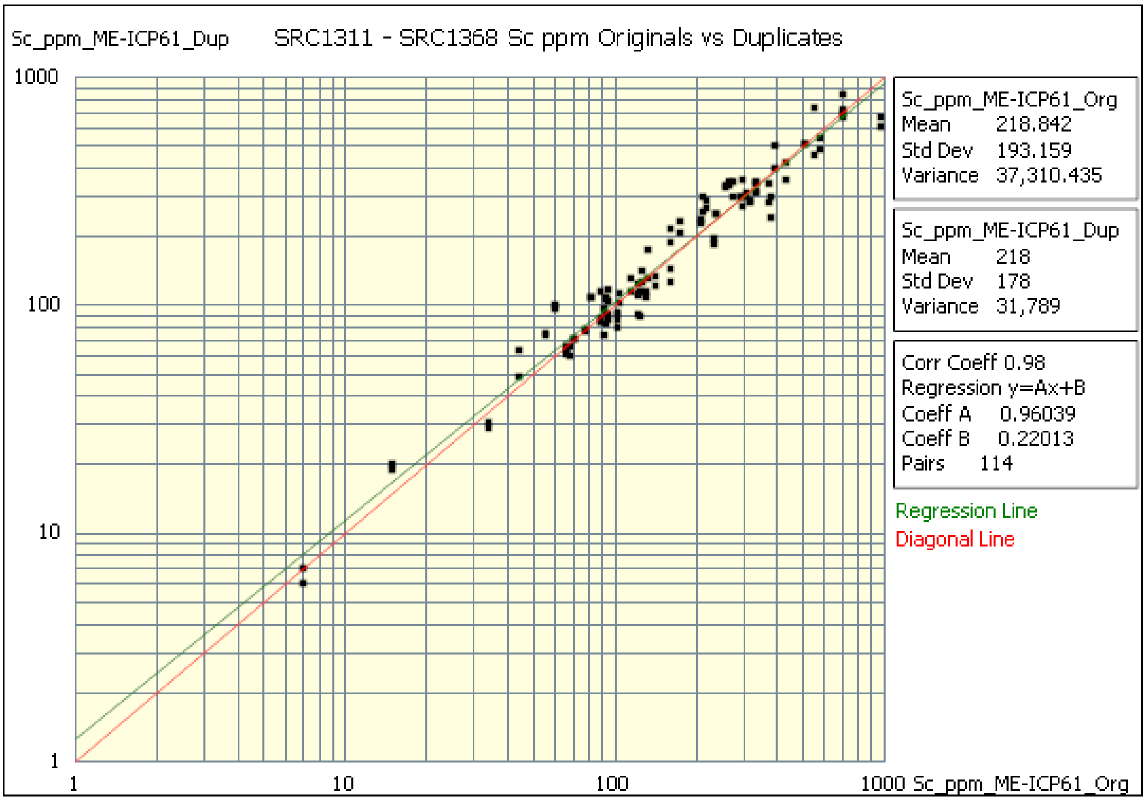


Figure 11-44: SRC1311 - SRC1368 Scandium PPM Originals vs Duplicates

11.7.6 Replicates

Laboratory replicate sample values were available for all the major drilling programs and this data was sourced from historic records as Microsoft Excel™ workbooks and imported into Micromine™ for initial analysis. The replicate data was imported into GeoAccess 2018 software and several tests to confirm laboratory repeatability were applied.

Original verses duplicate tests included:

- correlation coefficient
- diagonal line (1:1) verses regression line (bias test)
- QA/QC limits test at 10%, 20% and 30%.

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

SAC120 – SAC267 nickel replicate samples (see Figure 11-45):

- correlation coefficient:	1.00
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	4.3%
- statistical status:	statistically in control

SAC120 – SAC267 cobalt Replicate Samples (see Figure 11-46):

- correlation coefficient:	1.00
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	0.0%
- statistical status:	statistically in control

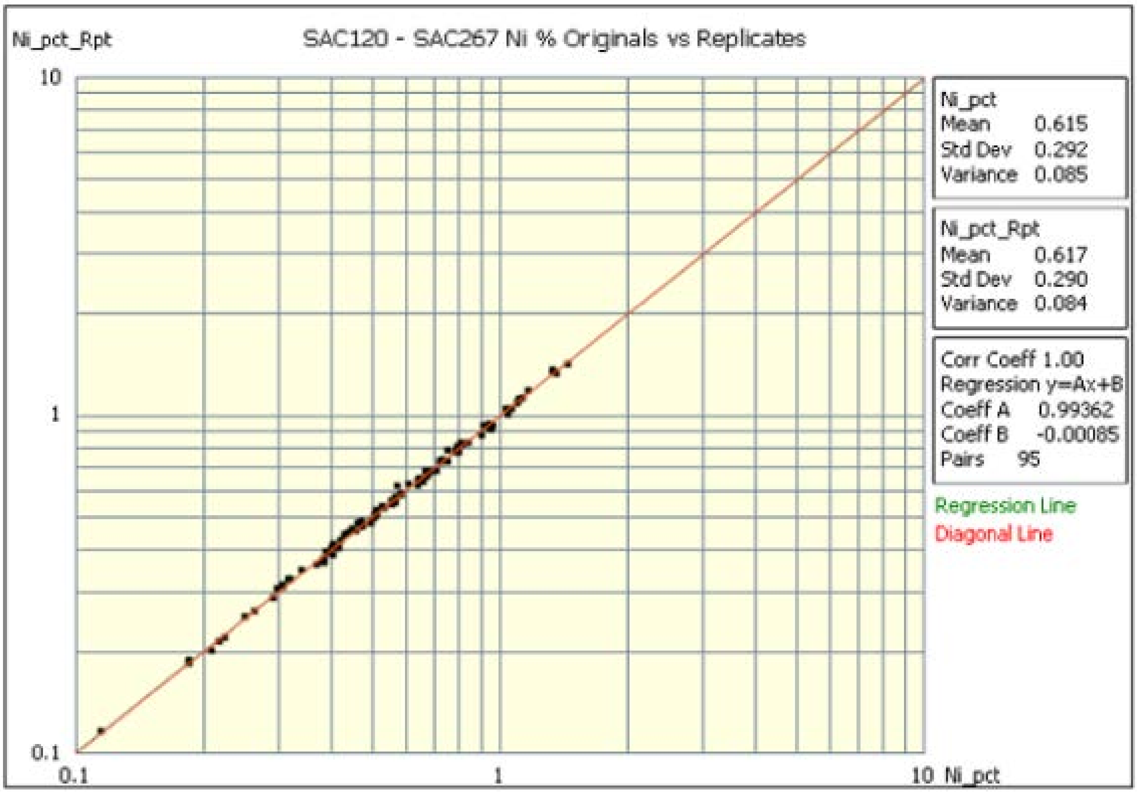


Figure 11-45: SAC120 - SAC267 Nickel % Originals vs Replicates

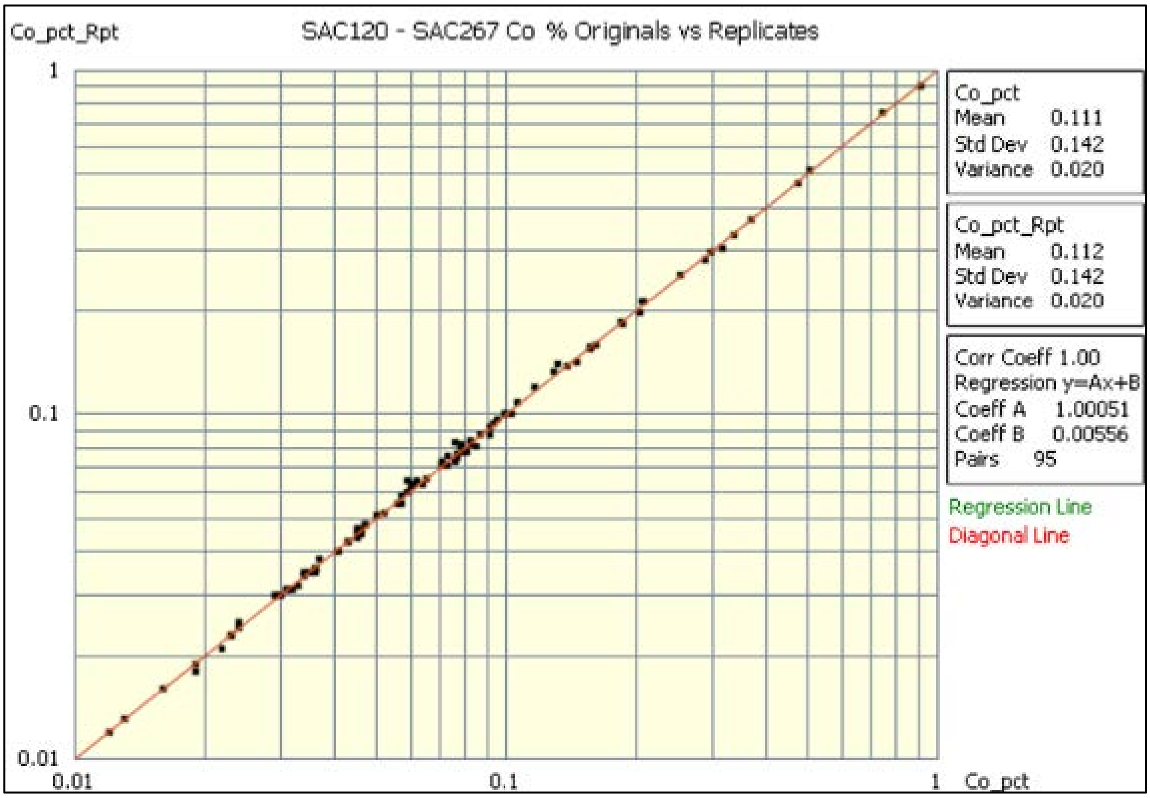


Figure 11-46: SAC120 - SAC267 Cobalt % Originals vs Replicates

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

SRC001 – SRC340 nickel replicate samples (see Figure 11-47):

- correlation coefficient: 1.00
- diagonal verses regression lines: no bias
- ±10% QA/QC limits: 0.8%
- statistical status: statistically in control

SRC001 – SRC340 cobalt replicate samples (see Figure 11-48):

1 - correlation coefficient: 2 1.00
3 - diagonal verses regression lines: 4 no bias
5 - ±10% QA/QC limits: 6 0.8%
7 - statistical status: 8 statistically in control

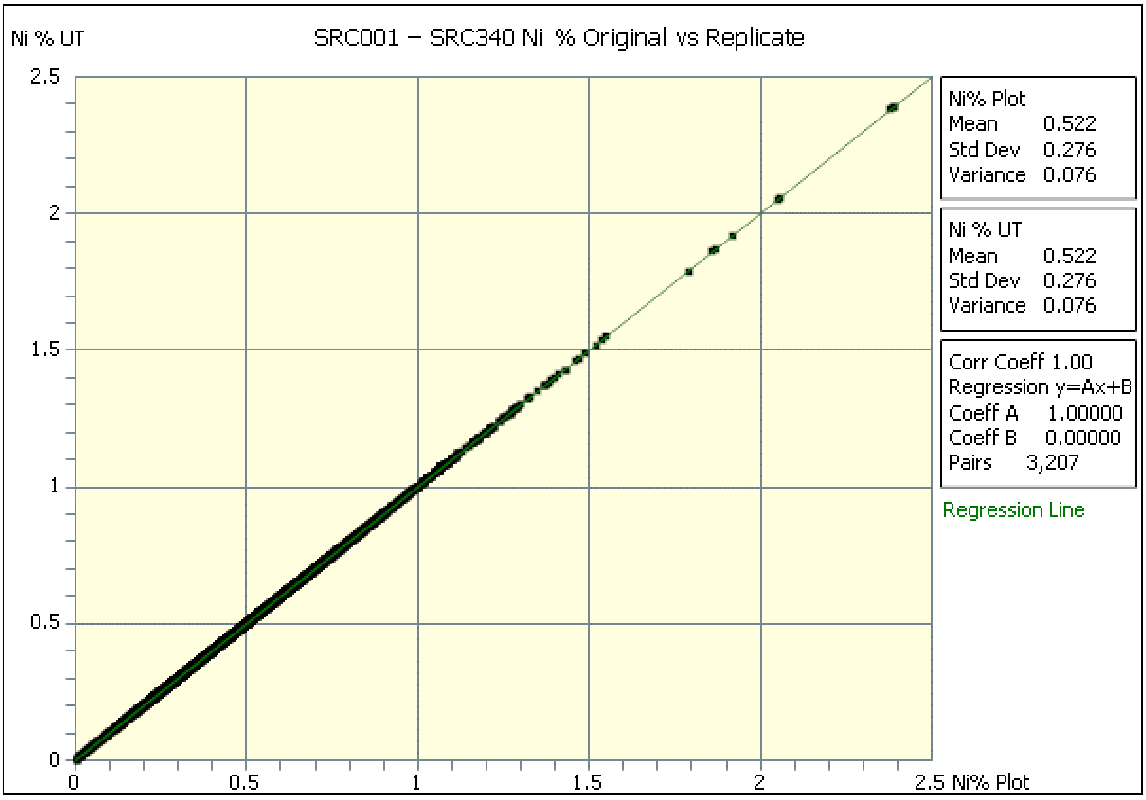


Figure 11-47: SRC001 - SRC340 Nickel % Originals vs Replicates

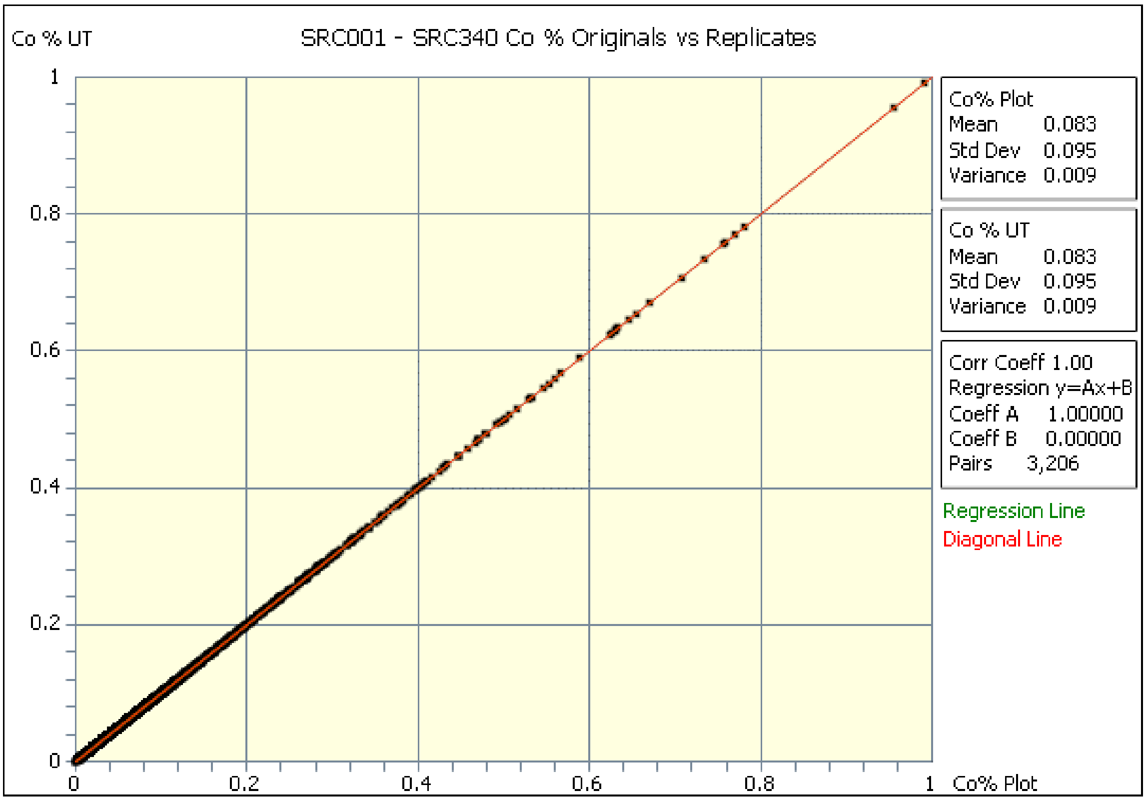


Figure 11-48: SRC001 - SRC340 Cobalt % Originals vs Replicates

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

SRC341 – SRC1076 nickel replicate samples (see Figure 11-49):

- correlation coefficient:	1.00
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	4.8%
- statistical status:	statistically in control

SRC341 – SRC1076 cobalt replicate samples (see Figure 11-50):

- correlation coefficient:	1.00
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	0.0%
- statistical status:	statistically in control

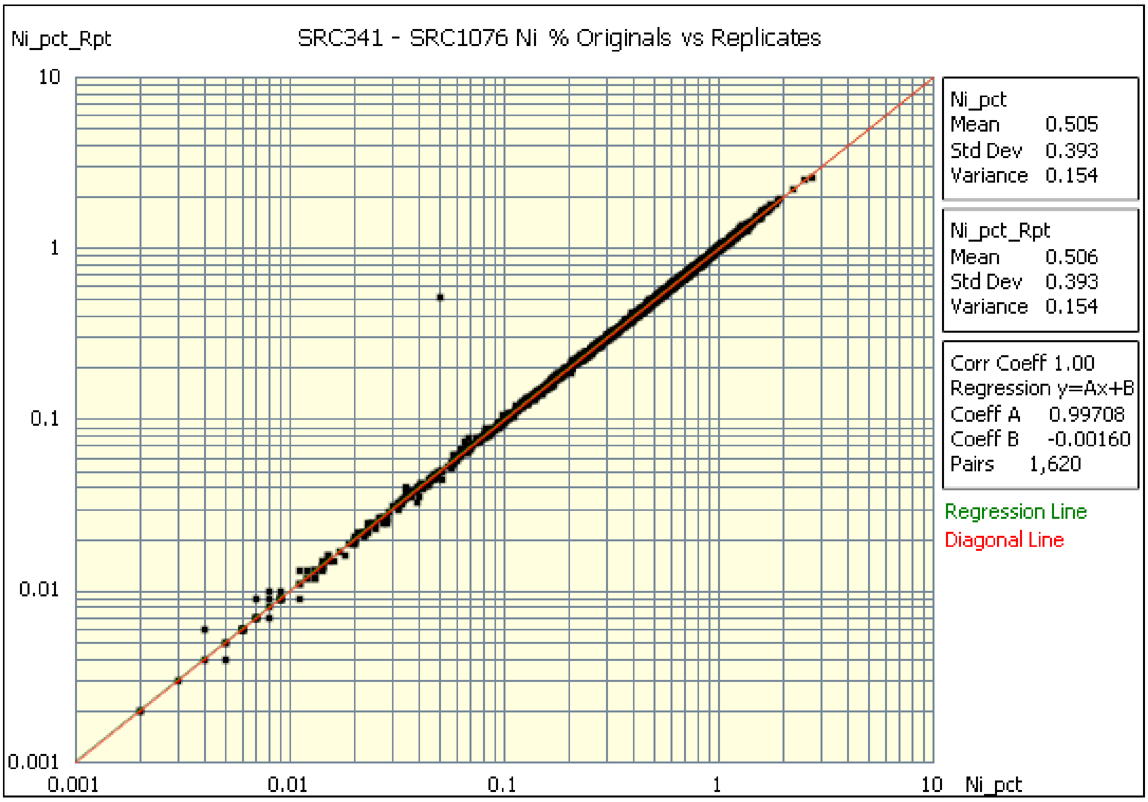


Figure 11-49: SRC341 - SRC1076 Nickel % Originals vs Replicates

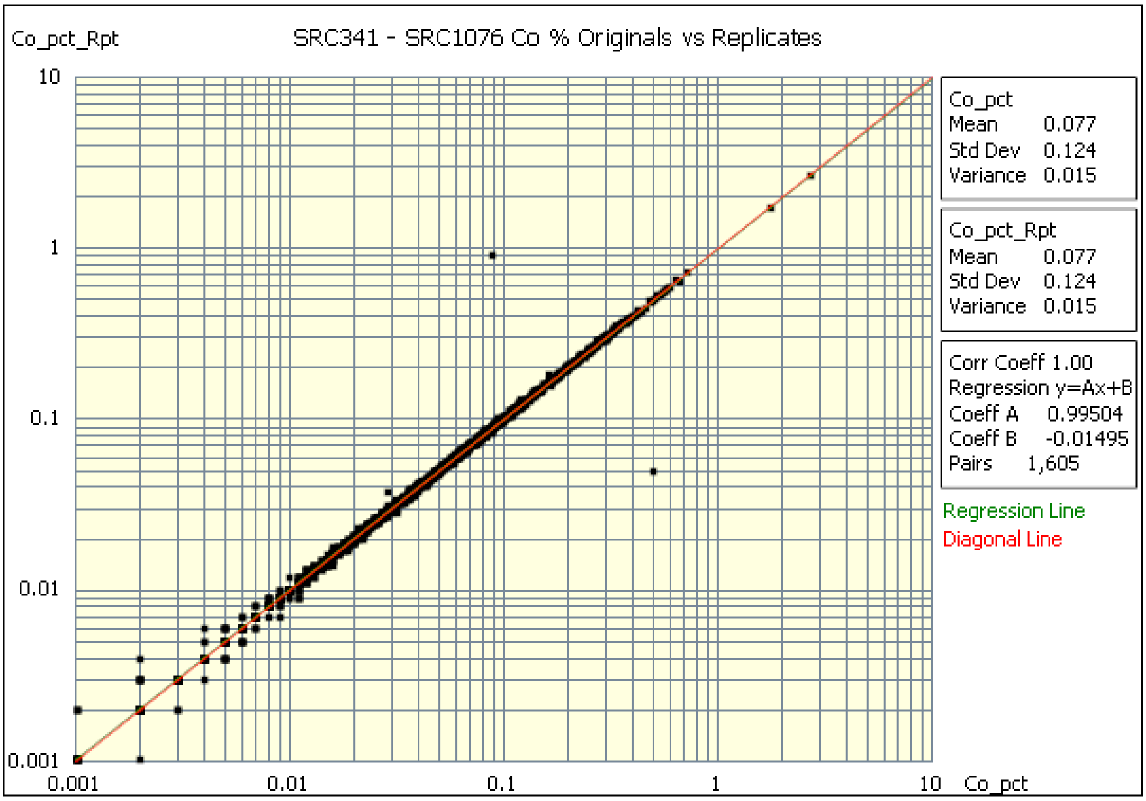


Figure 11-50: SRC341 - SRC1076 Cobalt % Originals vs Replicates

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

SRC1077 – SRC1251 nickel replicate samples (see Figure 11-51):

- correlation coefficient:	1.00
- diagonal verses regression lines:	no bias
- ±10% QA/QC limits:	0.0%
- statistical status:	statistically in control

SRC1077 – SRC1251 cobalt replicate samples (see Figure 11-52):

- correlation coefficient:	1.00
- diagonal verses regression lines:	very slight bias towards higher values
- ±10% QA/QC limits:	4.1%
- statistical status:	statistically in control

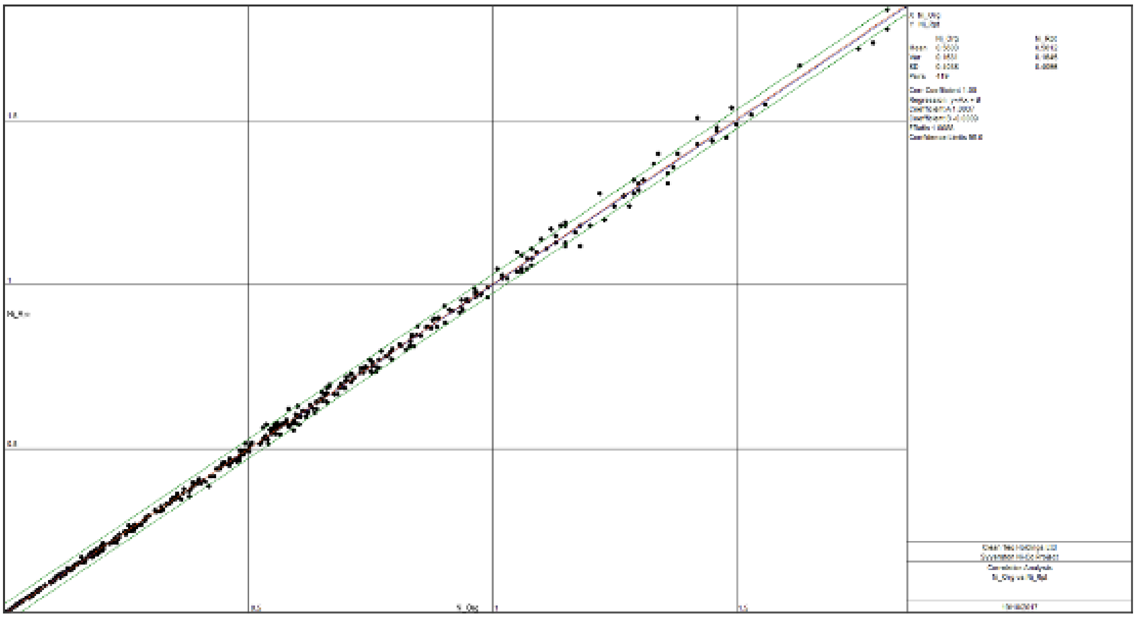


Figure 11-51: SRC1077 - SRC1251 Nickel % Originals vs Replicates

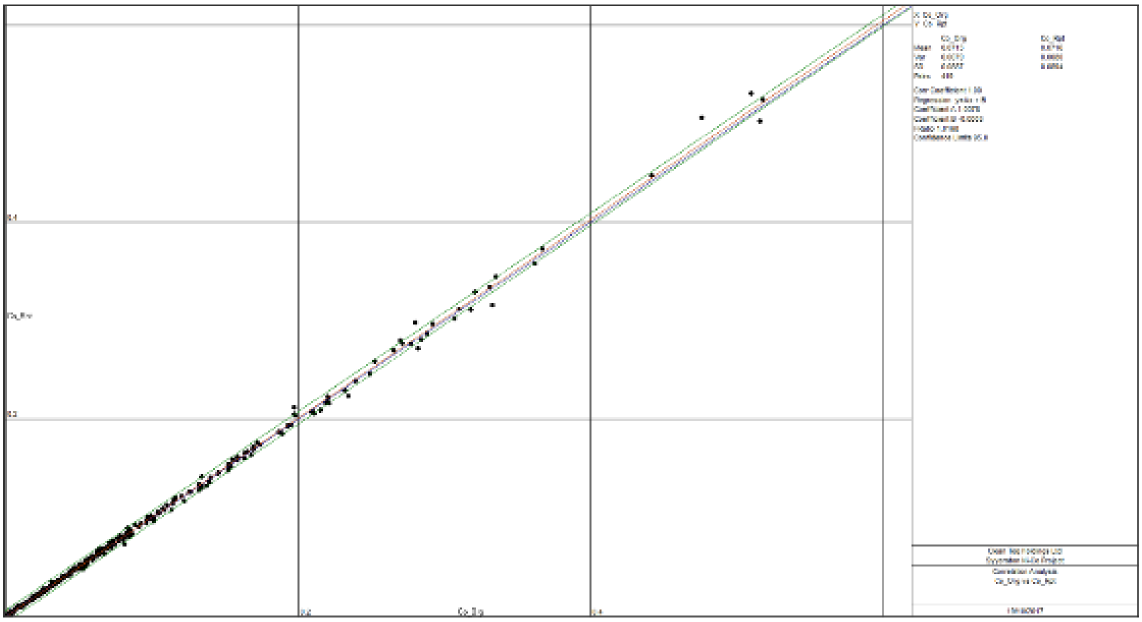


Figure 11-52: SRC1077 - SRC1251 Cobalt % Originals vs Replicates

SRC1263 - SRC1276 Drilling Program (August 2014)

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.

SRC1277 - SRC1310 Drilling Program (April 2015)

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.

SRC1311 - SRC1368 Drilling Program (November 2015)

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.

11.7.7 Check Assay Programs ALS vs Trace

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

SAC120 – SAC267 nickel ALS vs Ultra Trace check samples (see Figure 11-53):

- correlation coefficient: 0.96
- diagonal verses 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

SAC120 – SAC267 cobalt ALS vs Ultra Trace check samples (see Figure 11-54):

- correlation coefficient: 0.96
- diagonal verses 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

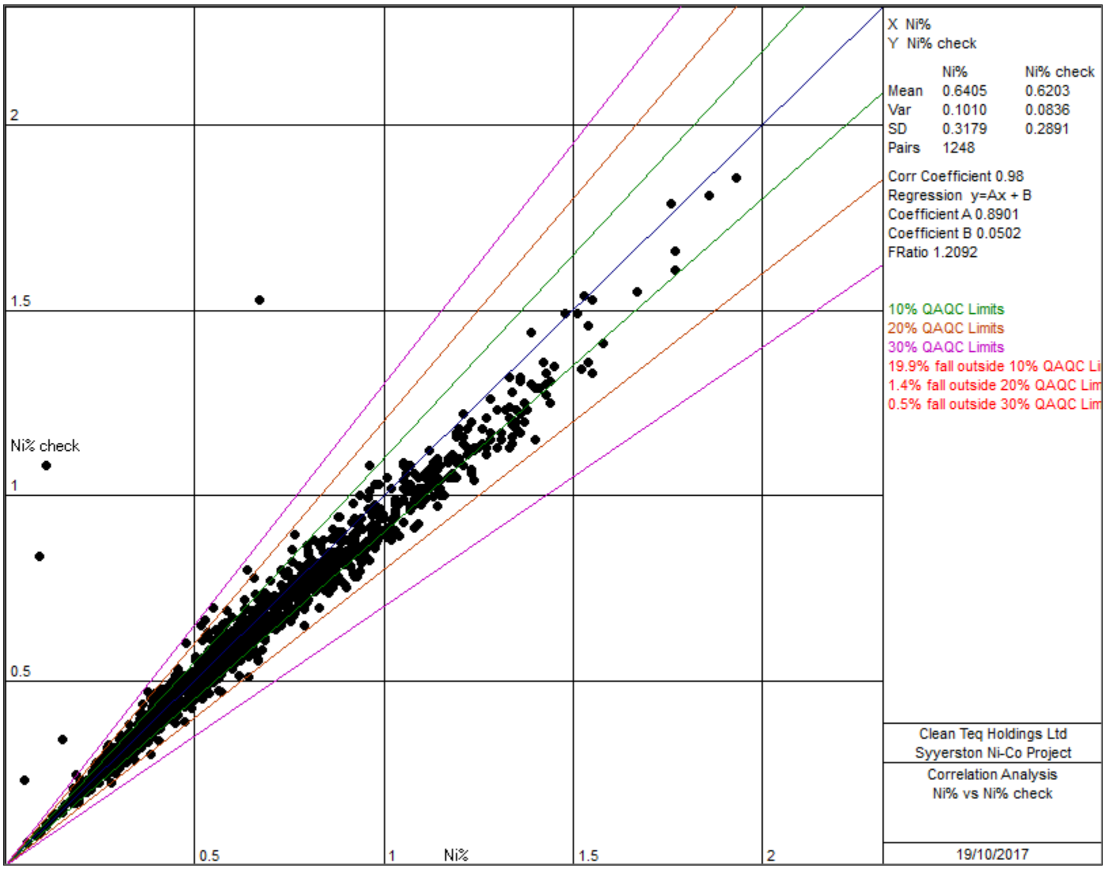


Figure 11-53: SAC120 - SAC267 Nickel % ALS vs Ultra Trace Check Assays

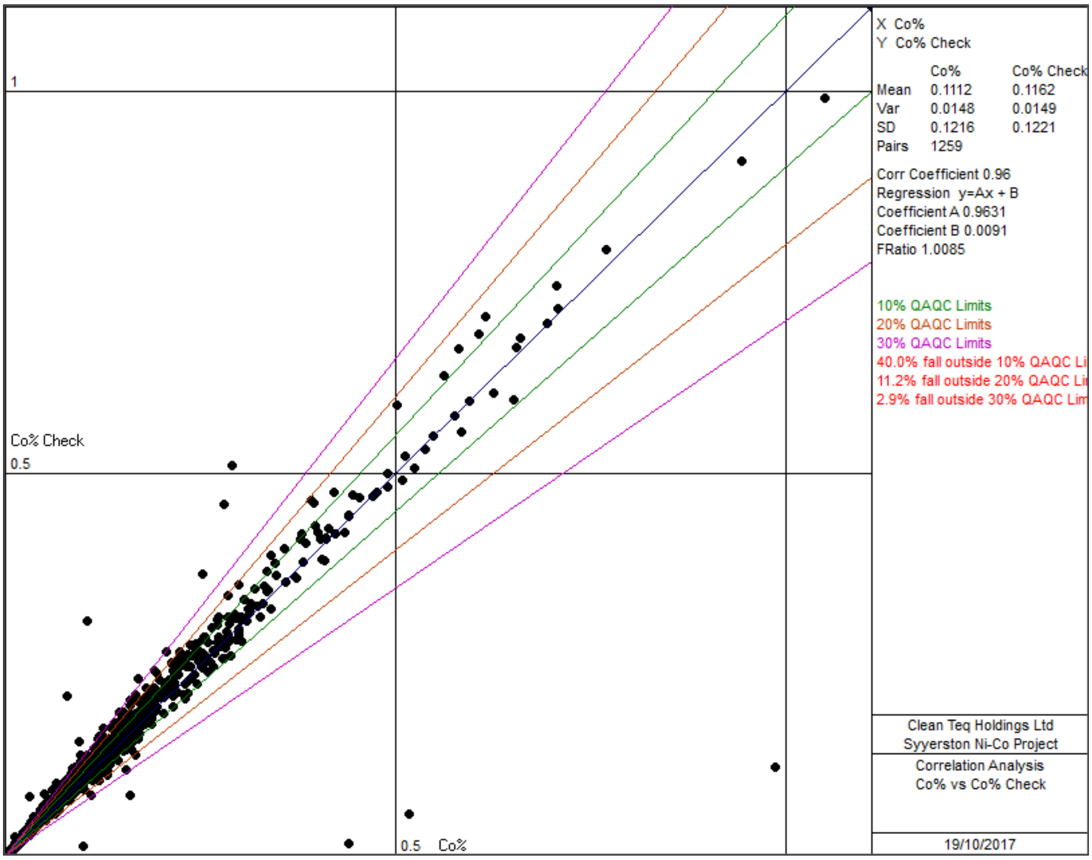


Figure 11-54: SAC120 - SAC267 Cobalt % ALS vs Ultra Trace Check Assays

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

SRC001 – SRC340 nickel ALS vs Ultra Trace check samples (see Figure 11-55):

- correlation coefficient: 0.96
- diagonal verses 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 cobalt ALS vs Ultra Trace check samples (see Figure 11-56):

- correlation coefficient: 0.96
- diagonal verses 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

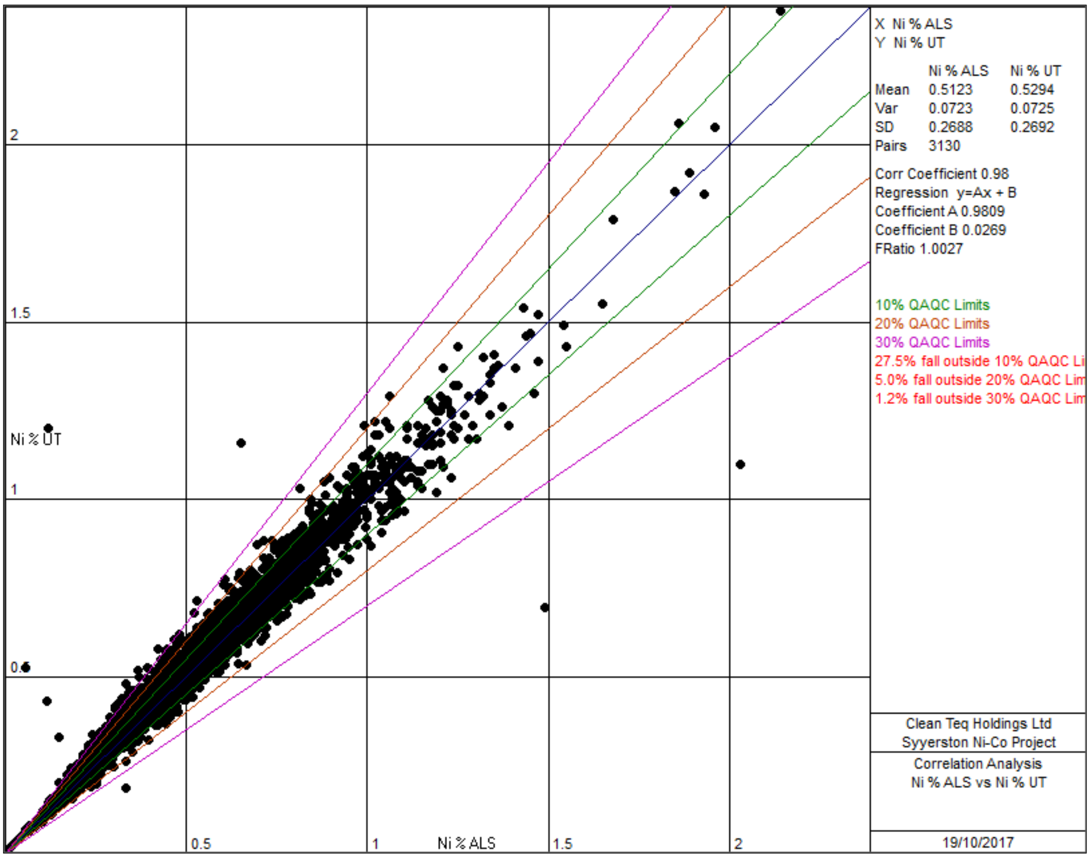


Figure 11-55: SRC001 - SRC340 Nickel % ALS vs Ultra Trace Check Assays

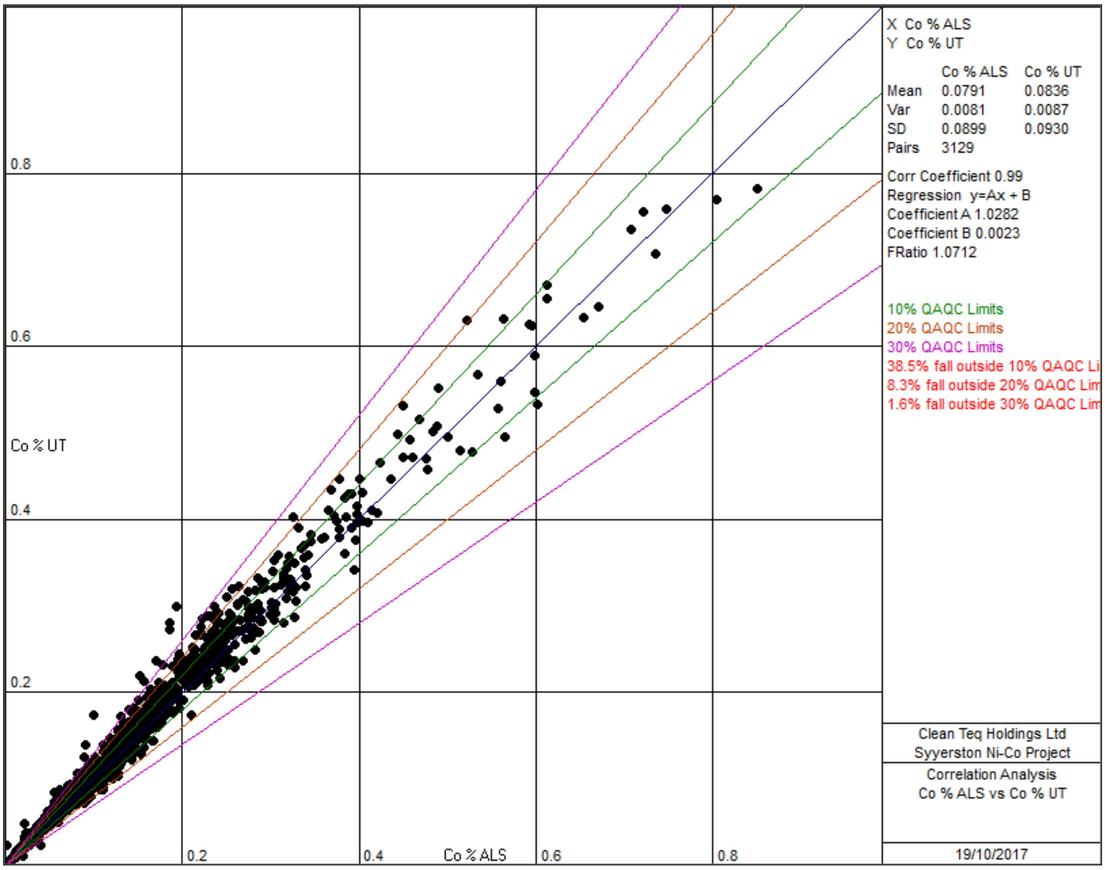


Figure 11-56: SRC001 - SRC340 Cobalt % ALS vs Ultra Trace Check Assays

11.7.7.3 SRC0341-SRC1076 Drilling Program (Aug98-Aug00)

SRC341 – SRC1004 nickel Ultra Trace Verses Genalysis Check Samples (see Figure 11-57):

- correlation coefficient:	0.99
- diagonal verses 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

SRC341 – SRC1076 cobalt Ultra Trace vs Genalysis check samples (see Figure 11-58):

- correlation coefficient:	0.99
- diagonal verses 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

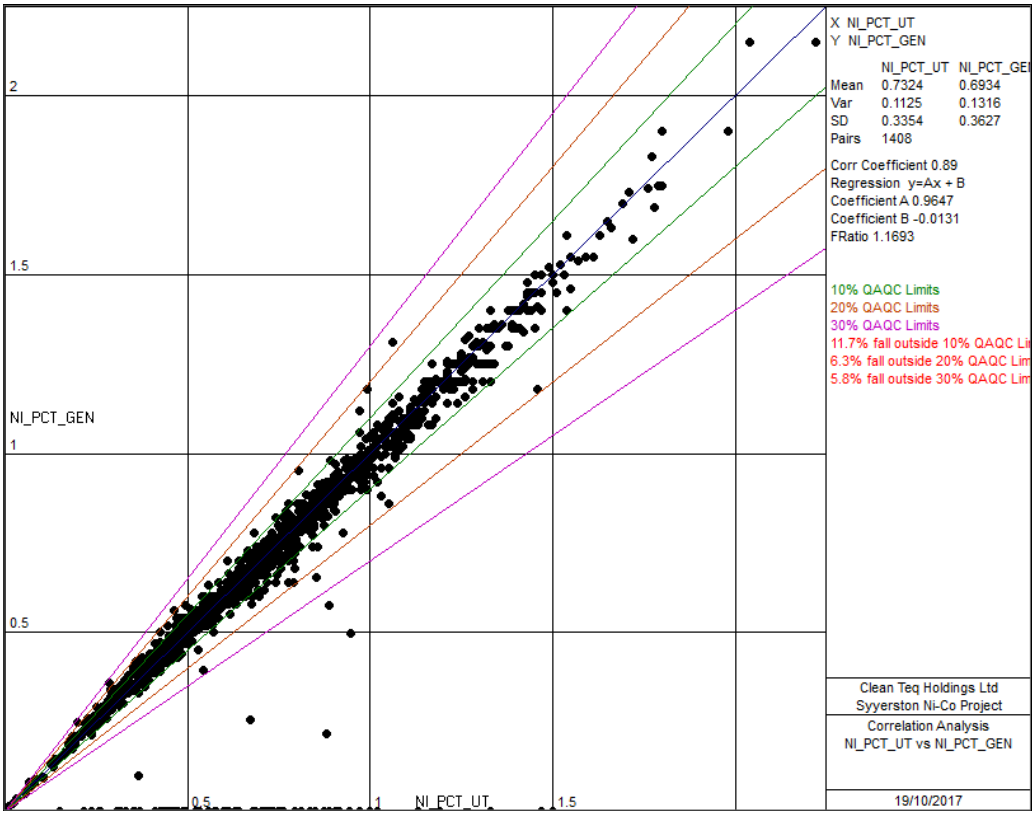


Figure 11-57: SRC341 - SRC1076 Nickel % Ultra Trace vs Genalysis Check Assays

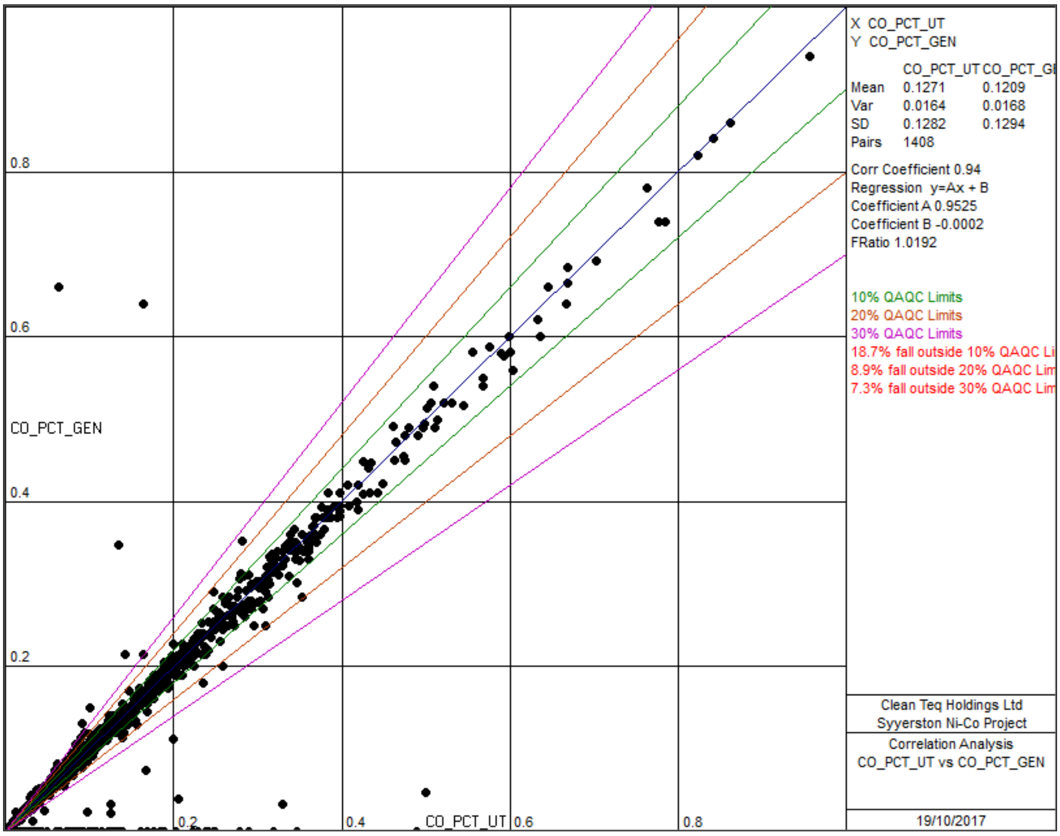


Figure 11-58: SRC341 - SRC1076 Cobalt % Ultra Trace vs Genalysis Check Assays

11.7.8 QAQA Procedures Comments

11.7.8.1 Shewhart Quality Control Charts

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

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 five standard values per batch.

The assay quality control charts display moderate variability with the laboratory values within the range of 2 SDs 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 one standard deviation.

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.

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

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 one standard per batch.

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

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.

11.7.8.2 Duplicates

Duplicate sampling programs were carried out on the SRC341-1076; SRC1263-1276; SRC1277-1310 (1 m and 2 m 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 nickel and cobalt were recorded as 0.98 and 0.99 respectively. There was a slight bias towards the duplicates. Whereas, the nickel values were within the ±10% QA/QC limits, the cobalt values reported 15.4% of the values reporting falling outside the QA/QC limits. The cobalt values were statistically ‘out-of-control’.

SRC1277 - 1310 Drilling Program

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

The 1 m 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 58-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 nickel, cobalt and scandium respectively. This indicated that for all three analytes that they were statistically 'out-of-control'. The duplicate sampling procedure required reviewing.

11.7.8.3 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 except for the SRC1077-SRC1251 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.

11.7.8.4 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 nickel and cobalt were 0.96. The nickel values display a slight bias towards the routine values, whilst the cobalt values are slightly biased to the check assay values.

The ±10% QAQC limits indicates that 19.9% of the nickel values and 40.0% of the cobalt 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 nickel, 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 nickel and cobalt were 0.96. The nickel values display no bias between the ALS and Ultra Trace data, however, the cobalt values display a slight bias towards the Ultra Trace check assays.

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

SRC341 - 1076 Drilling Program – Ultra Trace Verses 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 nickel was 0.96 and 0.99 for cobalt. The data set indicates that there is a slight bias in the higher values towards the Ultra Trace results for both nickel and cobalt.

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

11.8 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 ten RC drill holes from the historic resource definition drilling. A twin RC hole was positioned approximately 5 m diagonally from the previous RC hole in the northeast quadrant, whilst a diamond hole was to be drilled in a similar location in the southwest 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
- nickel % graph
- nickel % value
- cobalt % graph
- cobalt % value
- iron % graph
- silicon % graph
- aluminium % graph

- LATZONE hatch
- nickel and cobalt % interval.

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

Table 11-16: Custody Sampling Original vs Twin Hole Intervals (0.4% Nickel Cut-Off Value)

Original Hole	Original Hole Intersection	Twin Hole	Twin Hole Intersection
SRC 946	25m at 0.91% nickel and 0.111% cobalt	SRC 1418	30m at 0.86% nickel and 0.088% cobalt
SRC890	22m at 0.79% nickel and 0.119% cobalt	SRC 1419	21m at 0.76% nickel and 0.095% cobalt
SRC 742	31m at 0.83% nickel and 0.113% cobalt	SRC 1420	25m at 0.87% nickel and 0.168% cobalt
SRC 767	28m at 0.64% nickel and 0.076% cobalt	SRC 1421	26m at 0.67% nickel and 0.076% cobalt
SRC 774	19m at 0.65% nickel and 0.110% cobalt	SRC 1422	17m at 0.52% nickel and 0.172% cobalt
SRC 647	21m at 0.67% nickel and 0.127% cobalt	SRC 1423	18m at 0.47% nickel and 0.068% cobalt
SRC 1125	35m at 0.78% nickel and 0.156% cobalt	SRC 1424	31m at 0.83% nickel and 0.051% cobalt
SRC 445	31m at 0.78% nickel and 0.156% cobalt	SRC 1425	30m at 0.78% nickel and 0.149% cobalt
SRC 556	14m at 0.59% nickel and 0.117% cobalt	SRC 1426	25m at 0.48% nickel and 0.116% cobalt
SRC 597	17m at 0.55% nickel and 0.092% cobalt	SRC 1427	19m at 0.62% nickel and 0.144% cobalt

An assessment of the twin hole data was undertaken, and the key variables are summarised Table 11-17.

Table 11-17: Assessment of Twin Hole Data

Criteria	Assessment Result
Average interval difference (m)	-0.10
Average nickel % difference	-0.03
Average cobalt % difference	0.0
Average % difference interval	4.0
Average % difference nickel %	-4.5
Average % difference cobalt %	2.9

These results indicate that there are minor differences in the interval meters, nickel values and cobalt 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 Drill Hole Database

12.1.1 Database Structure

The Clean TeQ drill hole data is currently stored in a high level relational database management system (RDMS), that is, Microsoft SQL Server 2016™. The drill hole data is stored in a specific drill hole database model (CLQ_MODEL) initially configured when the Micromine™ Geobank™ database management program is first run (CLQGB_CONFIG). This process initiates the creation of various tables, which constitute the Geobank™ data model (GDM).

The Geobank™ program provides the ability to undertake a wide range of data activities using objects accessible using the Geobank™ Configuration Manager (see Figure 12-1). The assorted options in the Configuration Manager under the user interface enables various objects to be customized to suit the requirements of the project as the objects are derived from SQL scripts accessing the GDM database. In general terms, the principal objects available (but not limited to these objects), include the following:

- import objects
- export objects
- report objects
- QA/QC objects.

The specific objects are modified in the Configuration Manager by the database administrator and then transferred to the appropriate user profile in the Home tab (see Figure 12-2).

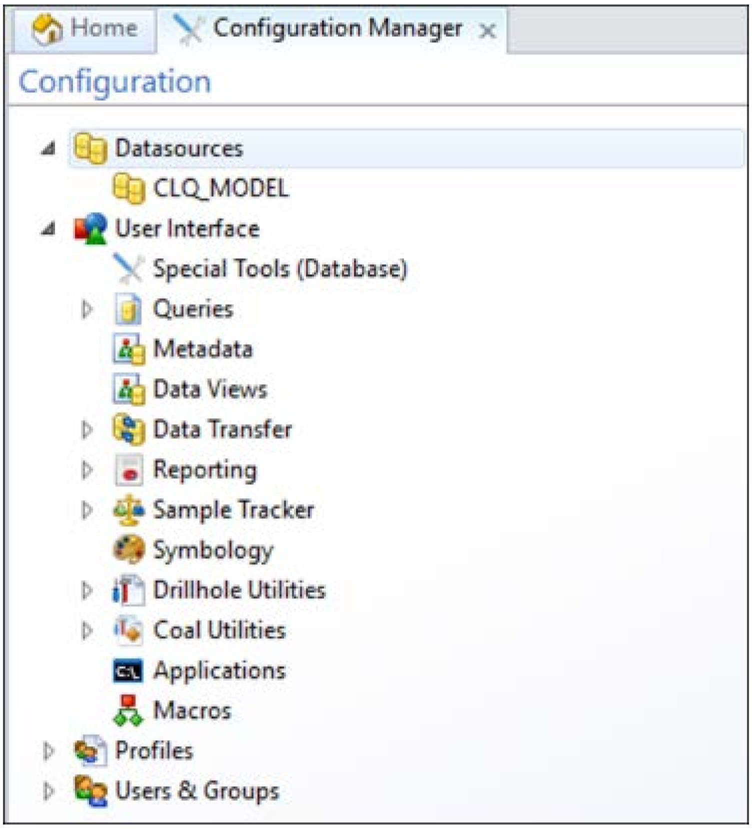


Figure 12-1: Micromine™ Geobank™ Configuration Manager Options

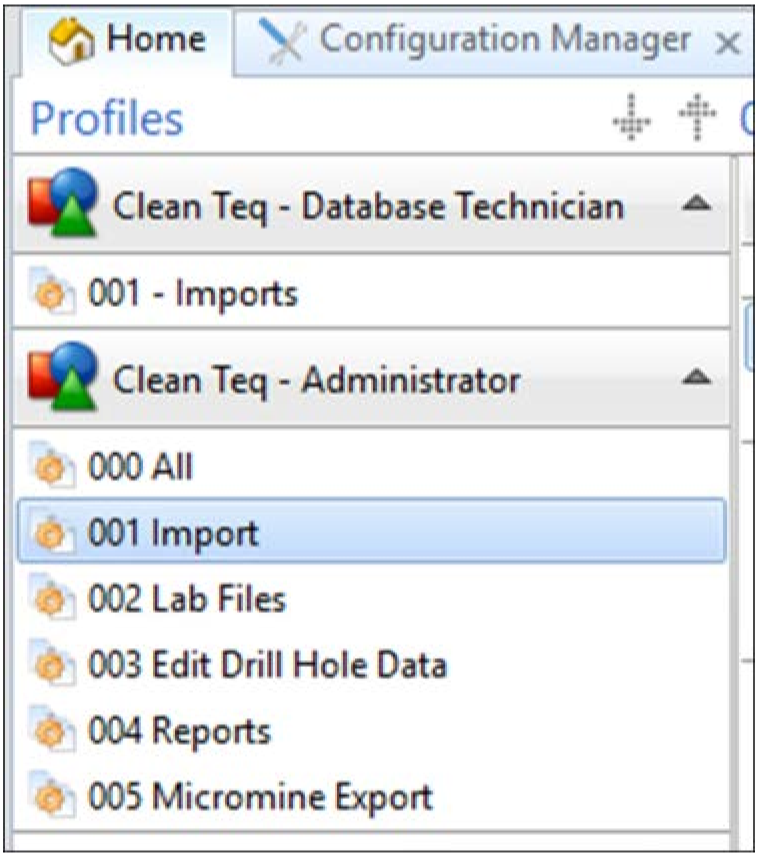


Figure 12-2: Geobank™ Home Tab with Customised Objects

The drill hole data is stored in the GDM in rigid tables for some of the data (GB_SITE, GB_SITE_SURVEY, etc.) whilst others such as the sample (GB_SAMPLE), the lithology (GB_LITHOLOGY) interval tables and the assay result (ST_RESULT) tables may be expanded to meet the requirements of the project.

The sample tracker module represents a significant advance in manipulation of the sample dispatch and receipt data as well as the merging of assay records. It also enables the recording of assay metadata including analytes, methods, units, limit of detection, etc for each laboratory for the subsequent validation of the imported assay records. This extensive module also keeps a track of the QA/QC data including duplicates, standards, blanks, etc as distinct from the routine assay samples.

12.1.2 Drill Hole Data Sources

The CLQ_MODEL drill hole database was established to store and validate the following principal types of data:

- drill hole location
- drill hole down hole surveys
- drill hole sample intervals
- drill hole geology intervals.

The initial data source for the location, downhole survey and sample interval data was from the Black Range Minerals NL Syerston_DB Microsoft Access™ database. This Microsoft Access™ database incorporated the following AC, RC, diamond and Calweld drilling programs:

- SAC120 – SAC267 inclusive
- SRC001 – SRC1076 inclusive

- SDD1 – SDD13 inclusive
- SCW1 – SCW9 inclusive.

The remainder of the resource definition RC drill holes (SRC1077-1251 and SRC1263-1368) were sourced from various Microsoft Excel™ workbooks prepared by Ivanplats Syerston Pty Ltd and Scandium21 Pty Ltd.

The drill hole location, downhole survey, sample interval and lithology records were exported from the Syerston_DB database as Microsoft Excel™ workbooks and imported into Micromine™. In Micromine™, the data was formatted to emulate the various tables in the GDM to ensure total compatibility, especially with respect to the data type. The Micromine™ data files were exported as Microsoft Excel™ workbooks before being saved as comma delimited text (csv) files.

The ad hoc import object in Geobank™ was used to import the drill hole location, downhole survey, assay interval and lithology interval data into the CLQGB_MODEL drill hole database.

The drill hole location data was recorded in the local grid, but a high proportion of the drill hole collars were surveyed and reported in AMG84 coordinates. The most recent drill holes were surveyed and reported in MGA coordinates. This necessitated the establishment of three survey coordinate instances in the database.

To validate the exiting coordinate records in the database, the survey reports from the registered surveyors Geolyse Pty Ltd (Geolyse) were imported into Micromine™ and load files for all three coordinate systems were created and ultimately imported into the database using the ‘ad hoc’ import object. The surveyed coordinates within the database were identified as either surveyed or calculated and using the AMG84, local and MGA - local grid transformation functions supplied by Geolyse, the three coordinate instances were fully populated.

To comply with the ‘chain-of-custody’ requirements for the importation of the laboratory assay records, the identification of the original sif or csv report files was required. If correctly documented, these assay report files contained the required metadata in the header to enable the imported records to be fully validated.

A high percentage of the original assay report files from the two principal routine laboratories (Ultra Trace and ALS) were identified in the archive records of the project with their original file date stamp. These files were copied to maintain the integrity of the original laboratory result files. In sample tracker, several receipt file formats were created to read the metadata in the header portion of the assay result files. Where the metadata was incomplete, (Geobank™ required the despatch number, laboratory job number, sample identification, result, element, units and method as a minimum requirement), then the missing component was edited within the result file.

As a prerequisite to importing the assay values, the analytes, methods, units, etc were recorded in the various sub-components in sample tracker. The various QA/QC types, as well as the standard details, are recorded in a further sub-component and applied to the sample rules in the import template if required.

Within the configuration manager, several receipt objects were created to reflect either a different receipt file format, or in the case of more recent drilling programs a unique import object, for each of these programs to ensure that they reflected each new batch for the purposes of the QA/QC monitoring.

The individual laboratory assay result files were imported separately using these receipt objects and the analyte, method, units and limit of detection were validated. Any records, which could not be resolved using the sample rules were flagged to be resolved interactively. These unresolved records were generally either standards or blanks and once these issues were fixed, the assay results were loaded, and an audit trail text and pdf load file were created.

12.1.3 Quality Assurance/Quality Control Report Objects

With all the historic and recent drilling program data imported successfully into the database, the functionality of the report objects facility built into the database can be customised to create various QA/QC reports e.g. Shewhart assay quality control charts (see Figure 12-3).

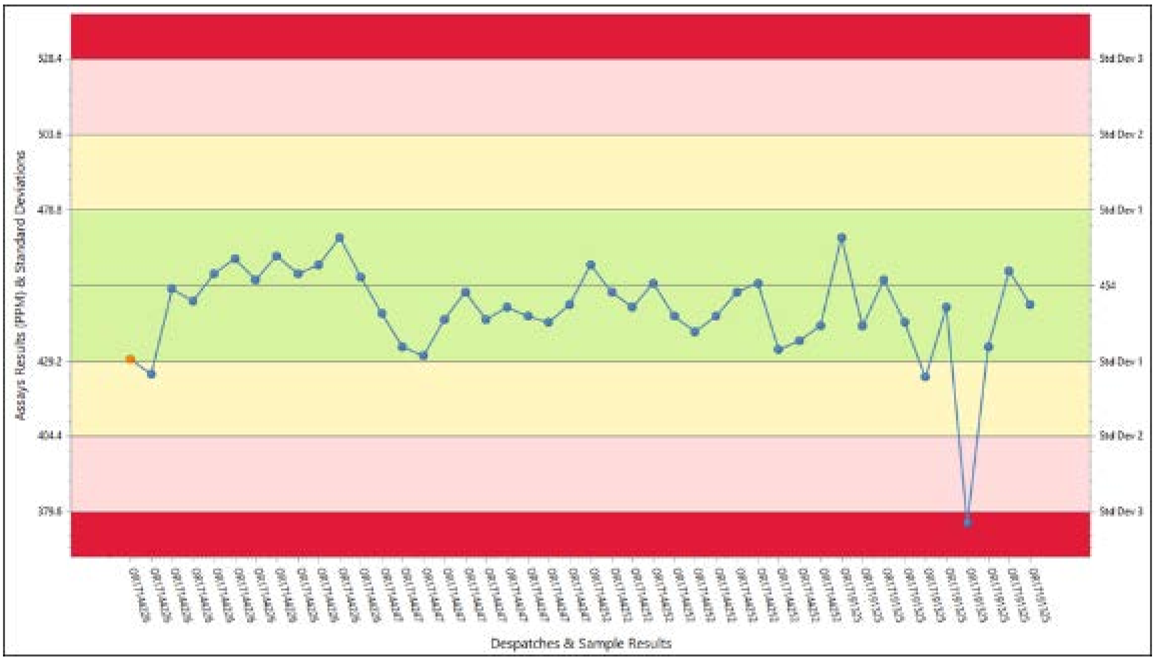


Figure 12-3: ALS Nickel (ME-ICP61) OREAS 45e Standard Quality Control Chart

The creation of Shewhart assay quality control charts for standards submitted with the routine samples, enables one or more batches to be monitored to determine whether the batch or batches are statistically in control and to respond to any errors or outliers promptly.

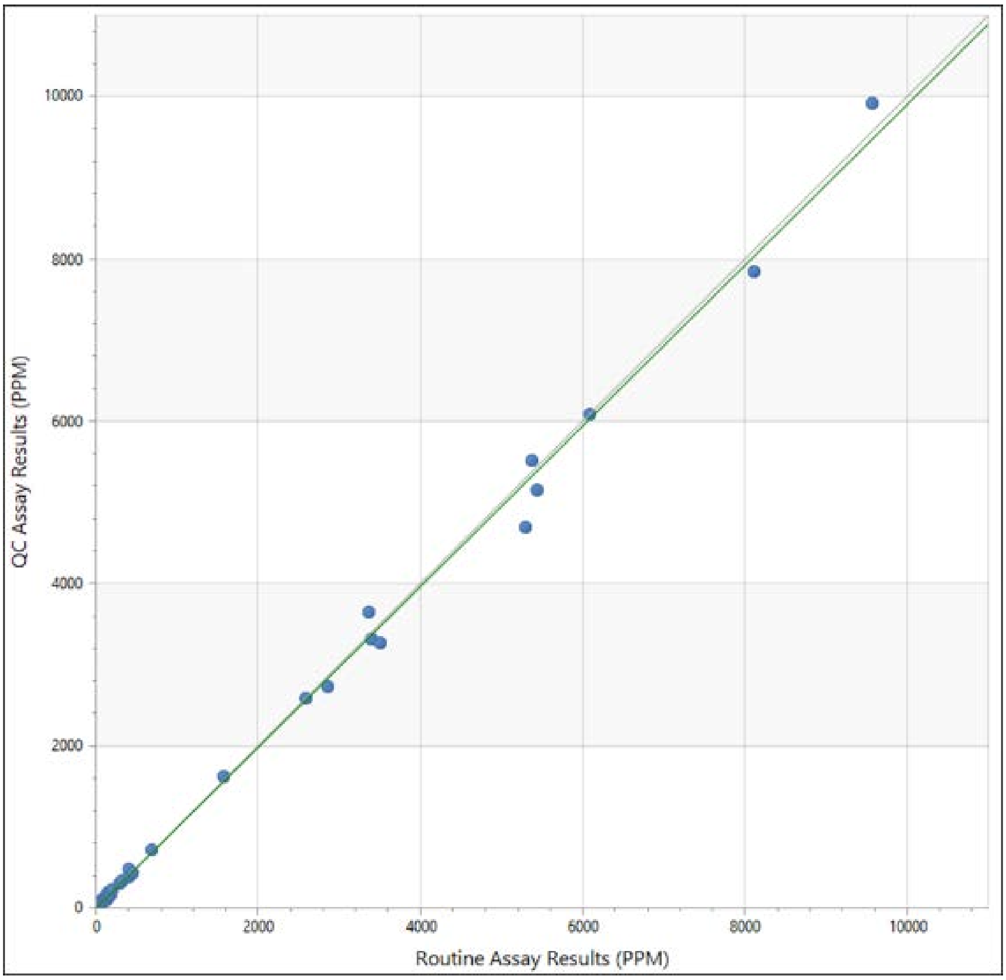


Figure 12-4: ALS Nickel (ME-ICP61) Routine vs Duplicate Scattergram

The creation of x-y charts or scattergrams of the routine verses the duplicate sample data establishes whether any bias exists between the two sets of values as determined by the linear regression line (green) (see Figure 12-4 above). If distinct outliers are present these samples may be identified and rectified by either re-sampling or re-assaying. The duplicate samples provide a means of determining whether the sub-sampling in the field and the sample preparation at the laboratory provide a representative sample.

12.1.4 Export Objects

Existing data views of the drill hole collars, downhole surveys, assays and lithology data were modified to suit the drill hole data specific to the project. This data was re-formatted into Micromine™ data file format and then exported into a nominated directory with standardised names, that is, collar, survey, assay and lithology.

Export objects were created to enable the QA/QC stored within the database to be exported to be evaluated in an external software package e.g. Micromine™ or Microsoft Excel™. The QA/QC data was exported in two separate batches, with one batch incorporating the standard and blank data, whilst the other batch included the duplicate data. The QA/QC included the full metadata relating to the samples including despatch number, job number, method, analyte, etc.

12.1.5 Coordinate Transformation

The highest-ranking coordinate system in the database is the local grid, however, the earlier hole programs were surveyed by DGPS using the AGD84 datum and AMG84 zone 55 projection. All the historic drill hole data was entered into one of the three coordinate systems (local, AMG84 and MGA) with transformations being undertaken prior to the records being imported into the database.

As all new DGPS surveys are to be carried out using the GDA94 datum and MGA zone 55 projection, then the GB_SITE_SURVEY records could be directly imported to populate the MGA coordinate instance. A coordinate transformation object was created to enable the remaining two coordinate instances (Local and AMG84) to be recorded within the database using the transformation parameters embedded in the coordinate systems module.

12.1.6 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

The proposed nickel cobalt plant (Figure 13-1) will be a pressure acid leach plant using similar technology and equipment currently being employed at other current operations but will incorporate a continuous Resin in Pulp (cRIP) ion exchange technology to replace counter current decantation prior to downstream refining requirements. The refinery will take nickel and cobalt rich solution (eluate) from the cRIP circuit and, using both solvent extraction (SX) and ion exchange (IX) processes, produce high purity nickel sulphate ($NiSO_4.6H_2O$), cobalt sulphate ($CoSO_4.7H_2O$) and scandium oxide (Sc_2O_3), as well as an ammonium sulphate (Amsul) by-product.

Figure 13-1: Schematic of the Process Plant

The plant has a design capacity of 2.5 Mtpa of feed to the pressure acid leach autoclave. The ore preparation plant has the capacity to treat a higher tonnage of plant feed and beneficiate the siliceous ore to reject ore with a lower mineralisation of nickel and cobalt, notably silica (quartz).

Extensive metallurgical testing and piloting on the extraction and recovery of nickel and cobalt, including variability testing on >100 composites of different ore lithologies, was completed by the two previous owners. During each of these testwork programs nickel and cobalt were the primary targets, with scandium also followed in the analyses. This work provided a relatively high degree of confidence on metal extraction using the High-Pressure Acid Leach (HPAL) process. In addition to this previous work, basic confirmatory sighter tests and pilot tests have been carried out by the current owners.

The more recent metallurgical testwork and processing development focus of Clean TeQ has been in applying the continuous Resin-In-Pulp (cRIP) technology to the HPAL discharge slurries. This has been a change to the original processing flowsheet and has been the focus of extensive testing and development. During the Definitive Feasibility Study (DFS), extensive testing of the flowsheet downstream of cRIP, specifically the nickel and cobalt solvent extraction and the final nickel and cobalt sulphate crystallisation process, has been undertaken, although they are not considered to be a novel technology. This has allowed the company to demonstrate that the process can produce high quality battery-grade nickel and cobalt products directly from the ore without the need for an intermediate product.

13.1 Processing technology

13.1.1 High pressure acid leach (HPAL)

The nickel, cobalt and scandium bound within the goethite can be solubilised from the solid matrix by acid leaching. Under the conditions present in the high-pressure acid leach (temperatures of about 250°C), the soluble iron species subsequently hydrolyse to insoluble basic iron sulphates, hematite and other iron products, and also produce sulphuric acid, a major saving in the operating cost. Soluble aluminium sulphates also hydrolyse to the basic sulphate salts with some acid production. This regeneration of acid is the major difference between HPAL and atmospheric leaching of laterite ores, which can also extract significant nickel and cobalt values. However, for atmospheric leaching, the leaching of nearly all the iron from the solids results in a high acid consumption and requires a high initial acid addition. Much less acid is regenerated and operating costs for acid are much higher.

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 where, currently, approximately 2 Mtpa of limonitic ore is processed by a HPAL in vertical brick-lined autoclaves, counter-current decantation (CCD), neutralisation using coral mud and mixed sulphide precipitation with hydrogen sulphide gas.

The HPAL route was further developed by Amax in the 1970's, who carried out an extensive pilot-scale testing 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 that is still used today. Although Amax never proceeded to commercial production, their testwork 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, near Kalgoorlie. Other Western Australian projects in the same area, 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 routes pre- and post-HPAL varied between the Western Australian producers, influenced by the nature of the ores, the quality of available process water, and selected final products (metals or metal compounds).

Murrin Murrin, which is still in operation, adopted the Sherritt process which involves precipitation of the metals as sulphides, re-dissolution in an oxidative leach with oxygen under pressure, followed by solvent extraction and hydrogen reduction. Processing at Bulong did not involve precipitation of the sulphide but used direct solvent extraction of the nickel and cobalt from the HPAL discharge liquor (after partial neutralisation and CCD), with the metals produced by electrowinning (electrode depositing). Cawse precipitated the nickel and cobalt as a hydroxide, originally redissolving the precipitate 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).

More recently, the Ravensthorpe project in Western Australia used enhanced pressure acid leach technology (similar to the Cawse flowsheet but also including a proprietary atmospheric leach process). Similarly, the Goro Project in New Caledonia utilises a process route similar to the Cawse design, while the Coral Bay (Sumitomo) and Ambatovy projects (Sherritt) utilise the Sherritt process, producing a mixed sulphide.

The HPAL processing technology is no longer new, now generally being considered to be in its fourth generation and is much improved by the experience gained in previous and currently operating plants, as demonstrated 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 shown by MCC's Ramu Project in Papua New Guinea but these challenges are well understood. The Ramu Project is now operating at throughputs exceeding design.

The development of HPAL for the extraction of scandium is also now widely accepted. Metallica Minerals completed a Pre-Feasibility study on the Sconi Project, 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, but generally lower than nickel and cobalt. The metallurgical testwork completed in the previous two feasibility studies on the Project, typically followed scandium, as well as nickel and cobalt.

13.1.2 Resin-in-pulp

Clean TeQ uses a proprietary ion exchange technology (Clean-iX®) for extraction and purification of metals and for 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, with further enhancement by Clean TeQ.

ARRICT 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. It 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 the atomic industry and other industries. The technologies have been adapted for processing gold-bearing, molybdenum, tungsten and other ores. ARRICT's continuous ion exchange technology for extracting metals from leached slurries and solutions has been used in many full-scale mining operations, including 22 uranium mines and 6 gold mines, mainly in the Former Soviet Union (FSU), Figure 13-2.

Figure 13-2: Continuous ion exchange plant with U-shaped columns in FSU (uranium)

In 2000, Clean TeQ obtained the exclusive 99-year 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 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 both batch and continuous Resin-In-Pulp (cRIP) technology:

- 2001-02: Development of cRIP 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 cRIP and desorption process for nickel and cobalt from laterite HPAL leached ores – several patents lodged. Development was with BHP Billiton (BHPB) for future laterite deposits.
- 2006: Development of extraction and purification of scandium from laterite ores.
- 2008: Licence signed with BHPB for nickel and cobalt technology.
- 2009: Uranium process for alkaline and hypersaline leach solutions – patent lodged.
- 2010: Development of cRIL (continuous Resin-In-Leach) process for gold thiosulphate leach solutions.
- 2011-15: Piloting for low grade scandium recovery from acidic sulphate and chloride TiO_2 process streams – patent lodged.
- 2015-2018: Large scale piloting of scandium extraction process on The Project ore. Lab-scale and pilot testing for nickel and cobalt extraction process on Sunrise ore.

Figure 13-3 is an overview of the Clean-iX® intellectual property developed and owned by Clean TeQ:

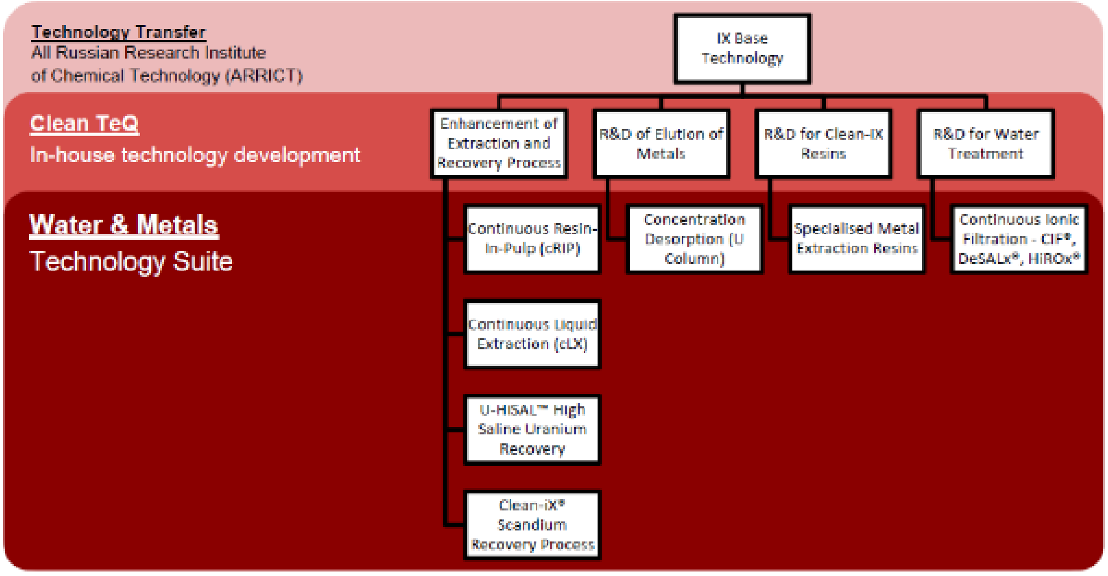


Figure 13-3: Clean-iX® intellectual property development

The application of cRIP for metals 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.

Typical nickel and cobalt recovery flowsheets use CCD 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 compared to the Clean TeQ Sunrise process including: high capital cost due to the difficulty in separating the liquid from the solids, lower recoveries due to solution losses in the CCD underflow, and higher capital and operating costs for re-leach of intermediate products and SX stages to refine to a saleable product.

Continuous RIP eliminates many of these issues by using solid ion exchange resin beads to directly contact with the leached slurry to extract over 99% of the target metals contained in the solution, without the need for CCDs. In this way, it is similar to the carbon-in-pulp (CIP) process for gold extraction. Ion exchange resins are ideal for recovery and concentration of lower concentration metals, which is the case in laterite processing plants, meaning that the relative chemical costs and plant size is reduced compared to SX.

13.1.3 Downstream Refining

The Clean TeQ Sunrise Refinery will take nickel- and cobalt-rich solution or eluate (also containing scandium) from the continuous cRIP circuit and produce high quality final products of nickel sulphate, cobalt sulphate and scandium oxide, using a number of well-known and proven metallurgical processes to remove impurities and separate the metals. These processes will include pH adjustment and precipitation, solvent extraction and ion exchange. As the refinery will start with high tenors of metals in solution and incorporate a number of recycle streams to minimise metal losses, the overall recovery of metals in the refinery will be high (estimated at 99.9% nickel and 99.6% cobalt).

Nickel and cobalt separation and refining from intermediate products has been undertaken for many years, using chemical precipitation, solvent extraction and ion exchange processes. Major refineries using these processes are located at Norilsk Nickel (Russia), Jinchuan Group (China), Glencore (Sudbury, Canada and Nikkelverk, Norway), Huayou Cobalt (China), Queensland Nickel (closed) and Murrin Murrin (Australia).

13.2 Historical metallurgical testwork

The historical metallurgical testwork carried out on both The Project ore and, in the case of cRIP, other nickel laterite projects, as well as more recent extensive batch and pilot testwork carried out during 2016 to 2018, was used as the basis for the establishment of the process design criteria for the 2018 DFS.

The definitive metallurgical batch and pilot testwork completed by the two previous owners, including variability testing of the different ore lithologies, has provided a solid basis on which to establish the design criteria for the current project, particularly upstream of the cRIP process. The two metallurgical studies 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 analyses. 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 (now SGS Minerals) and Ammtec (now ALS Metallurgy) in Perth, Western Australia, and Hazen Research in Denver, Colorado, USA, with SNC-Lavalin's (project engineer) 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 providing process design criteria for the plant.

The key process units were piloted by Hazen Research and encompassed the following areas: High Pressure Acid Leach; counter current decantation; solution pre-reduction, neutralisation; and sulphide precipitation. The intermediate nickel/cobalt sulphide was leached; followed by iron removal from pregnant liquor; cobalt-zinc solvent extraction; zinc solvent extraction; nickel hydroxide precipitation and tailings neutralisation. Typically, the continuous runs went for 5 to 12 days.

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; HPAL variability testwork; and rheology testwork.

13.2.2 Feasibility study update testwork (Ivanplats Syerston, 2005)

Testwork for the Feasibility Study Update (FSU) by Ivanplats Syerston commenced in November 2004 and was completed in March 2005. The work was carried out at Lakefield Oretest laboratories in Perth, Western Australia, 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 Lakefield Oretest. They encompassed HPAL; counter current decantation and tailings neutralisation; with pre-reduction and neutralisation of the leach solution; followed by 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 characterisation, HPAL batch testwork, rheology testwork; agitation testwork; and flume beaching testwork on tailings.

13.3 Clean TeQ Feasibility Study 2018

13.3.1 Metallurgical Sample Validity

Clean TeQ have reviewed the samples selected for both the 2000 Feasibility Study and 2005 Feasibility Study Update testwork and concluded that both sets of metallurgical variability samples and bulk composites aligned with the samples tested within the 2018 definitive feasibility testwork and that all samples are a true reflection of ore to be processed within the mine plan. It can therefore be reasonably assumed that the metallurgical performance demonstrated in the previous testwork is a fair indication of the expected metallurgical performance. This has been verified by the metallurgical performance of the new samples tested since the start of 2015 by Clean TeQ.

13.3.2 Ore preparation

An ore preparation program of testwork was carried out at ALS Metallurgy in Perth, Western Australia in 2016 to confirm and update the process design criteria for the ore preparation circuit and to provide 20 tonnes of autoclave feed ore for HPAL testing

The size by size analysis indicated that the ore is very fine with the majority of the nickel and cobalt in the ore present in the sub-500 µm size fraction, with the coarser ore (+500 µm) showing elevated grades of silicon. Based on previous studies, the milling circuit was run open circuit to reject coarse silica from the ore blend.

Other ore preparation batch testwork undertaken as part of the ore preparation program included ore scrubbing to upgrade the ore, and gravity concentration to remove a chromite concentrate. This was to help reduce the potential for excessive wear on the HPAL feed pumps and to reduce the potential for downstream process issues generated by the formation of chromium(VI) in the autoclave discharge leach slurry. While the ore scrubbing did not prove to be beneficial due to target metal losses in the oversize discard fraction, the inclusion of a gravity circuit for chromium recovery could produce a middling fraction (1.5% mass) containing up to 10% chromium. This may be beneficial in reducing both the wear to feed pumps and potential for chromium(VI) formation in the subsequent autoclave discharge slurry. In addition, there may be added benefits in possible platinum recovery. However, it was not considered essential for the current plant design as the chromite-bearing ores are not an issue in the initial years of the project. Further testwork in this area is planned in the future.

13.3.2.1 Pressure Acid Leach Feed Thickening

In the 2005 Feasibility Study Update, Outokumpu (Perth) conducted dynamic thickening tests on composite one and composite three feed samples. The dynamic tests conducted on samples milled to a top size of 180 μm, indicated that 48% w/w solids was readily achievable; but the actual operating density may require reduction at times depending on the ore slurry viscosity.

Rheology testwork undertaken at Lakefield Oretest for the 2000 Feasibility Study indicated that the Sunrise ores exhibited variable rheology between 45%w/w and 48%w/w solids, with the yield stress often increasing with time (shear thickening). This will be overcome in the plant design by the inclusion of a shear thinning loop for the pressure acid leach feed thickener underflow, which has been shown to decrease yield stresses in the ore, and the latest technology in thickened HPAL feed pumping. This, coupled with a coarser top size target of 500 µm and elevated temperature (80^{o}C) suggest that the project will be able to manage to consistent underflow solids exceeding 45% w/w in operation. This work has been further confirmed in the 2018 pilot testwork where the coarser top size product of 500 μm has produced up to 52%wt solids with yield stress values below 100Pa.

The key process design criteria developed from the program are summarised in Table 13-1.

Table 13-1: Ore Preparation Key Process Design Criteria

Criteria	Measurement
Maximum Design mass rejection capacity	30%
Mass Rejection rate – goethite ore, Si < 15%	0%
Mass Rejection rate – siliceous goethite ore, 15% < Si < 23%	18%
Mass Rejection rate – siliceous goethite ore, Si > 23%	27%
Nickel, cobalt elemental recovery to product – goethite ore, Si < 15%	100%
Nickel, cobalt elemental recovery to product – siliceous goethite ore, 15% < Si < 23%	94%
Nickel, cobalt elemental recovery to product – siliceous goethite ore, Si > 23%	93%
Pressure acid leach feed top size, F100	0.5mm
Gravity concentrate mass recovery	<0.7%
PAL feed thickener underflow throughput (PAL feed)	2.5 Mtpa
PAL feed thickener underflow solids density (design, dynamic throughput)	45% w/w
PAL feed thickener underflow solids density (nominal)	48% w/w
PAL feed thickener underflow nominal yield stress	< 75Pa

13.3.3 High pressure acid leach testwork

Previous testwork for the high-pressure acid leach process for the Project provided the basis for the process design of the pressure acid leach circuit. A review of historical variability samples confirmed that approximately 45% of samples tested produced conditions likely to generate chromium(VI) in the autoclave. This confirmed that addition of a reducing agent to the autoclave feed will be ore dependent and will need to be added as required in operation. In 2016, Clean TeQ completed some batch testwork to confirm the potential for sulphur addition at 0.75kg/t as the reductant to decrease the formation of chromium(VI) in the autoclave.

In the 2018 DFS, pilot pressure acid leach testwork was undertaken at ALS Metallurgy in Perth. The pilot program was plagued by a number of mechanical issues, but under stable operating conditions produced nickel and cobalt leach extractions of 91.1% and 90.4% respectively. This was due to a significant reduction in acid addition to 200 kg/t, compared to 260 kg/t in batch testwork, due to a focus on achieving a final free acidity of 40-45g/L post flash, i.e. 30 g/L in the final compartment. This methodology was flawed resulting in the lower than expected leach extractions of nickel and cobalt.

Following the pilot plant, a second program of 20 batch tests utilizing a 4L autoclave was initiated using the remaining ore, to confirm the leach extractions at a target acid addition and to provide leach slurry for downstream cRIP testwork. The twenty batch tests using the increased acid addition of 260kg/t, which was determined based on the head grade analysis, resulted in achieving average nickel and cobalt extractions of 96.2% and 95.2% respectively. These results validated the acid requirement of 260 kg/t which is based upon the acid consumption equation derived from previous pilot and variability testwork results and solution chemistry.

From the results of the 2000, 2005 and 2018 testwork, the key process design and performance criteria have been developed (Table 13-2).

Table 13-2: Pressure Acid Leach Key Process Design Criteria

Criteria	Units	Measurement
PAL operating temperature (nominal)	ºC	250
PAL operating pressure (nominal)	kPa	4,500
PAL feed solids density (design, dynamic throughput)	% w/w	45
PAL feed thickener underflow solids density (nominal)	% w/w	48%
PAL residence time (minimum)	minutes	79
PAL residence time (nominal)	minutes	87
Free acid (pre-flash)	g/L	40 – 45
Pressure Acid Leach Extraction		
Nickel	%	95.7
Cobalt	%	95.0
Scandium	%	91.0
Acid addition (average)	kg/t	258
Acid addition (maximum)	kg/t	300
Acid addition (minimum)	kg/t	150

13.3.4 Pre-reduction and partial neutralisation

Previous testwork was completed on solution neutralisation in the 2000 Feasibility Study and the 2005 Feasibility Study Update where the counter current decant overflow was pre-reduced for the management of chromium(VI) followed by pH adjustment to pH 2.5 prior to feed the nickel cobalt refinery.

In the 2018 Definitive Feasibility Study, the counter current decant circuit has been removed and replaced by a total slurry pre-reduction step for the management of residual chromium(VI) followed by a partial neutralisation step where the slurry pH is increased to 4.0 to 4.2 prior to cRIP treatment.

The management of chromium(VI) will be by both sulphur addition to the pressure acid each feed (to undertake the majority of reduction in the autoclave) and the addition of sodium meta-bisulphite (SMBS) to the pressure acid leach discharge to act as a trim for final control of the oxidation-reduction potential (ORP) of the slurry to ensure that the chromium (VI) is reduced to chromium(III) with minimal excess reductant added. Excess reductant would result in excessive iron(II) formation.

In 2017, testwork was conducted at ALS Metallurgy (Perth) to confirm the leach slurry reduction of chromium(VI) to chromium(III) and to determine the parameters for the pre-reduced leach slurry pH increase to pH 4. This was followed by a mini-pilot plant operation to produce partially neutralised leach discharge for small scale cRIP nickel and cobalt testwork. At the time of testwork the use of sodium hydrosulphide (NaHS) was being considered and was used for both bench scale and pilot plant testwork. SMBS is proposed for the commercial flowsheet, and batch testwork with this reagent has been completed to confirm its suitability for use in the application. The use of SMBS will be further validated during continuous piloting commencing in July 2018. Both sulphur and SMBS were used successfully at Cawse for the reduction of chromium in the autoclave and in the discharge liquors after CCD, so no issues are foreseen in its use in the current flowsheet for pre-reduction of the chromium(VI) in autoclave discharge slurry.

Partial neutralisation testwork was completed at both batch and pilot scale on the pre-reduced slurry. At pilot scale, the process involved limestone addition to four aerated tanks in series at 70ºC, providing a total residence time of six hours. Under stable operating conditions, the process was successful in lowering the concentrations of iron to less than 50 mg/L, aluminium to less than 100 mg/L, and chromium to less than 20mg/L in the slurry, while nickel and cobalt losses by precipitation as hydroxides averaged less than 4%.

The key design and performance criteria are summarised in Table 13-3.

Table 13-3: Partial Neutralisation Key Process Design Criteria

Criteria	Measurement
Pre-reductant agent	SMBS 350*g/L
Pre-reductant stages	1, pipe reactor
Pre-reduction target ORP	<570mV
Partial neutralisation stages	6 x 60 minutes, agitated tank
Partial neutralisation air addition	4.2Nm^3/m^3 feed
Partial neutralisation terminal pH	4.0 – 4.2
Partial neutralisation discharge temperature (maximum)	100ºC
Neutralisation agent	limestone, 35%w/w
Limestone utilisation	95%
Partial neutralisation extent	-
Nickel co-precipitation	3.16%
Cobalt co-precipitation	3.75%
Scandium co-precipitation	100%
Aluminium tenor	<100mg/L
Iron tenor	<50mg/L
Chromium tenor	<20mg/L
Partial neutralisation solids top size, P_{100}	0.4mm

*Dosage of SMBS established from batch testing only. Pilot dosages to be determined in July 2018.

13.3.5 Clean TeQ cRIP testwork

13.3.5.1 Historical testwork

Nickel, cobalt and scandium cRIP extraction testwork on laterite ores from The Project and other projects and operations has been undertaken. From 2004-2008, Clean TeQ undertook a development program with BHP Billiton for the application of cRIP technology for nickel and cobalt extraction and purification from laterite ores. During this time, from laboratory scale up to 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 cRIP recovery of scandium from Sunrise ore (Figure 13-4). This testwork used high grade scandium ores from outside the current project, and 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 Sunrise 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

13.3.5.2 2018 Feasibility study cRIP testwork

The Project was modified for the 2018 DFS to recover nickel and cobalt directly from partially neutralised leach slurry, utilising ion exchange (IX) technology, without the need for prior solid-liquid separation. Clean TeQ's proprietary Clean-IX® technology utilises an ion exchange resin to selectively extract nickel and cobalt from the leach slurry using the continuous Resin-In-Pulp (cRIP) process.

Nickel and cobalt adsorb more strongly onto the imino-diacetic acid (IDA) chelating functional resin in a dilute sulphuric acid matrix (pH of ~ 4) than many other metals. The order of magnitude for loading of divalent metals onto the IDA ion exchange resin is:

$Ni^{2+} > Co^{2+} / Zn^{2+} > Fe^{2+} > Mn^{2+} > Ca^{2+} > Mg^{2+}$

However, in the presence of trivalent cations such as Al^{3+}, Fe^{3+} and Sc^{3+}, the IDA resin will load these ions before Ni^{2+} and Co^{2+}. Therefore, the pH of the slurry in the partial neutralisation circuit must be adjusted to pH 4 to precipitate these metals and minimise the concentrations of their ions in solution. Copper (Cu^{2+}) also has a higher affinity for adsorption on the IDA resin and is managed by partially precipitating it out of solution at pH 4 in partial neutralisation.

A testwork program was conducted during the 2018 DFS which focused on confirming and demonstrating the technology, its application to the Project, and to develop and validate the flowsheet configuration and design criteria.

The testwork program consisted of:

- assessment of alternative resins for the extraction of nickel and cobalt from the partially neutralised Clean TeQ Sunrise PAL discharge slurry, and the desorption of the metals back into a concentrated solution
- pilot plant operation of the proposed cRIP adsorption and desorption circuits on fresh partially neutralised PAL slurry discharge derived from ore obtained from the Project to support design parameters for the process design criteria
- mini-pilot plant operation of cRIP adsorption and desorption circuits on fresh partially neutralised PAL slurry to demonstrate optimised adsorption and desorption operation to support design parameters.

A total of five commercially available resins were tested for adsorption and desorption characteristics using both synthetic and pilot plant generated PAL discharge slurry neutralised to pH 4. The mechanical strength of the resins was also tested using the Russian Ball Mill (RBM) test which evaluates the hydromechanical strength of the resin when subjected to impact and abrasion forces simulating those experienced in the cRIP process. The test is widely used by resin manufactures and Russian technology developers and operators in evaluating resins for the cRIP application. Following this evaluation, the LANXESS resin TP209XL, and Purolite resin MtS9301, were chosen for further batch and pilot plant testing.

13.3.5.3 Nickel Cobalt Pressure Acid Leach/Continuous Resin in Pulp Pilot Plant

The continuous cRIP pilot plant was performed at ALS Metallurgy in Perth. The cRIP circuit used ion exchange resin to extract nickel and cobalt from partially neutralised PAL slurry (adsorption), screen out loaded resin from the barren slurry, then wash and elute the resin with eluant (dilute sulphuric acid) to produce a nickel and cobalt-rich eluate (desorption), and finally rinsing the barren resin prior to recycling it back to the adsorption circuit.

During the pilot plant operation, the barren resin was returned to the adsorption circuit in the protonated (H^+) form, to Pachuca 10 where limestone was added to neutralise the acid produced when these H^+ ions were exchanged for metal ions were adsorbed onto the resin. Because of this, the value metals nickel, cobalt, and scandium were found to re-solubilise to different extents in the cRIP circuit. This was driven by a shift in the equilibrium between the soluble metals and their precipitated hydroxides, as well as the recycling of desorbed resin returning from the desorption circuit in the protonated (H^+) form. Nominally, the extent of re-dissolution of the metals, as measured, were nickel (20%), cobalt (20%) and scandium (40%).

As this phenomenon is dependent on the balance between protonated (H^+ form) and unprotonated (Ca^{2+} form) barren resin being returned to the circuit, the re-dissolution of value metals will be pH-dependent and will need to be balanced against re-dissolution of the deleterious metals, especially iron and aluminium.

In the commercial cRIP operation, a neutralisation Pachuca will be installed after the desorption resin wash column to convert approximately 75% of the desorbed resin from the H^+ form to the Ca^{2+} form. There will need to be a balance between ensuring protons are available for the re-dissolution of the value metals, but not the deleterious metals. Therefore, the extent of re-dissolution of these metals may differ from the pilot plant results. As such a lower limit on the pH of the adsorption circuit tail of >3.7 is desirable to enable dissolution of target metals and limit that of aluminium and iron. This will be confirmed in pilot plant trials due to commence in July 2018.

The key findings from the cRIP pilot plant were:

- nickel and cobalt extraction from liquor to the resin using cRIP exceeded 99.5%
- low residual nickel (< 20 mg/l) and cobalt (<5 mg/l) remained in the barren solution
- nickel and cobalt co-precipitated in partial neutralisation is partially resolubilised (20% nickel and 20% cobalt) and extracted to the ion exchange resin in the cRIP adsorption circuit
- scandium extraction from the precipitated solid phase to the ion exchange resin indicated a potential to achieve up to 40% of PAL solubilised scandium recovery to the nickel cobalt eluate.

13.3.5.4 Nickel Cobalt PAL/cRIP Mini-pilot Plant

A further nickel cobalt PAL/cRIP mini-pilot program was initiated to provide additional process design data for the Project. The work was conducted in two phases, with the first stage (adsorption) at ALS Metallurgy in Perth, and the second stage (desorption) at Clean TeQ in Melbourne.

Adsorption

The main objective of the nickel cobalt pressure acid leach/cRIP mini-pilot plant was to confirm performance of the continuous resin in pulp (cRIP) process, in terms of adsorption chemistry, via a 5.5 day long continuous operating run. The mini-pilot plant represents a much more controlled circuit for resin advance, hence greater definition around adsorption and desorption process chemistry can be achieved. A diagrammatic representation of the cRIP mini-pilot plant is shown in Figure 13-5.

The testwork confirmed the IDA resin adsorption affinity for cRIP feed slurry is typically as follows:

Nickel >> Zinc > Cobalt > Manganese > Calcium, Magnesium

Figure 13-5: cRIP mini-pilot plant adsorption skid showing process stream movement

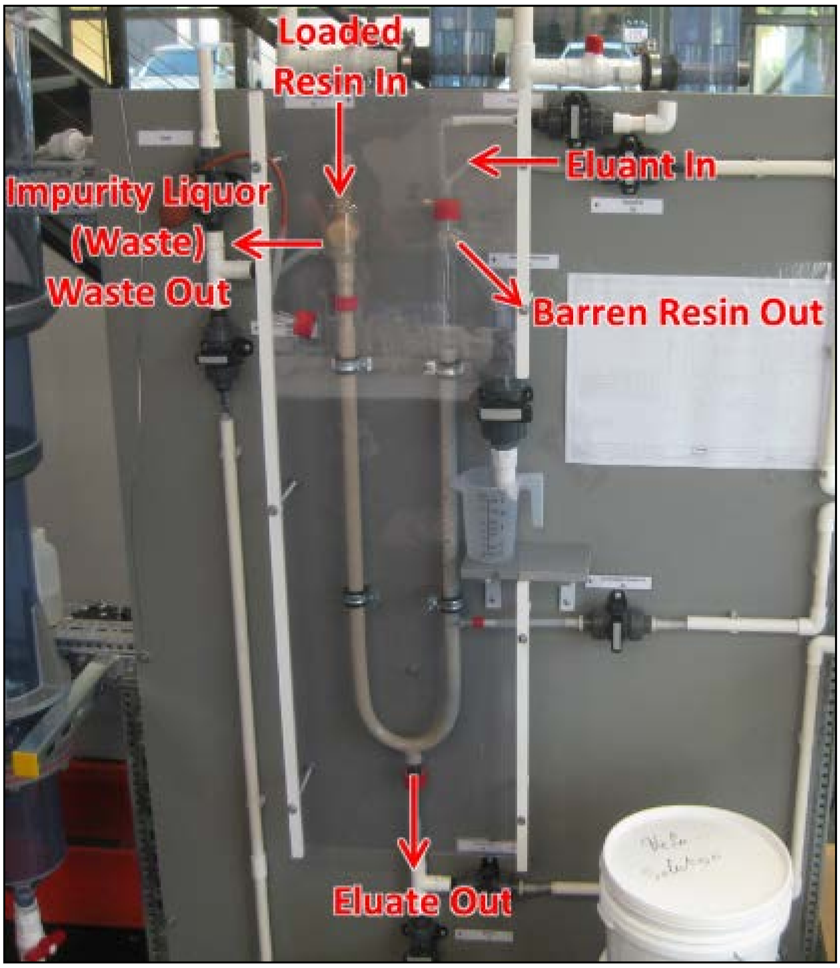
Desorption

Loaded resin from the mini-pilot plant testwork was collected and sent to the Clean TeQ laboratory in Melbourne where desorption tests were undertaken. Loaded resin was processed through the mini-pilot plant U-column, which was able to

- “super-load” or saturate the loaded resin with target metals and thereby deliver an enriched nickel cobalt eluate, and
- crowd impurities, including calcium, magnesium and manganese, from the resin and thereby deliver an upgraded nickel cobalt eluate.

Figure 13-6 provides an overview of the 800 mL U-column test rig. Loaded resin enters the top of the left-hand side (LHS) of the U-column and eluate (dilute sulphuric acid) enters the top of the right-hand side (RHS) of the column. Impurity liquor (waste) exits at the top of the LHS and barren resin exits from the top of the RHS of the U-column. The eluate containing the value metals exits from the base of the column. Nickel and cobalt losses to the impurity liquor (waste) represent a recirculating load of target metal back to the adsorption circuit.

Figure 13-7 provides a visual account of the desorption front progressing down the left-hand side of the U-column, prior to a resin advance being initiated (a manual operation on the test apparatus used).



Note: Impurity Liquor = Desorption Waste and Barren Resin = Resin in the H^+-form.

Figure 13-6: U-column general arrangement

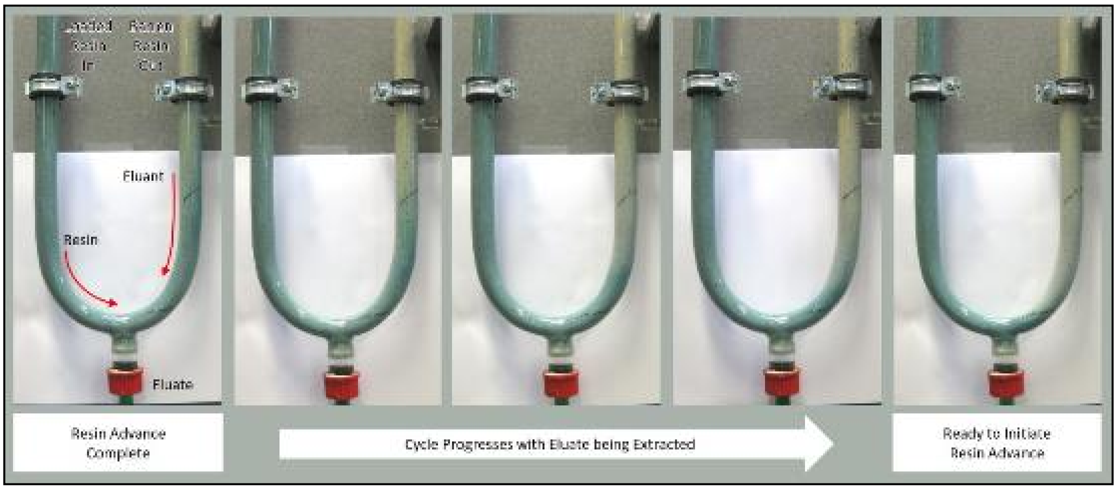


Figure 13-7: U-column desorption front between consecutive resin advances

From the results of the mini-pilot plant adsorption and desorption tests, the key design and performance criteria were developed (Table 13-4).

Table 13-4: Nickel cobalt cRIP key process design criteria

Criteria	Measurement
Resin functional group	Imino Diacetic Acid (IDA)
Number of adsorption stages (Pachuca)	10
Adsorption Feed pH range	4.0-4.2
Adsorption Discharge pH stage 10	>3.7
Metal Extraction	
Nickel tenor in cRIP discharge slurry	<20mg/L
Cobalt tenor in cRIP discharge slurry	<5mg/L
Scandium	100%
Loaded resin capacity (Ni + Co + Zn + Cu)	51g/L_{wsr}
Desorption vessel	U-column
Desorption residence time	10 hours
Eluant (nominal)	100g/L H_2SO_4
Desorption Waste Elemental Rejection	
Calcium	65%
Magnesium	95%
Manganese	50%
Eluate free acid	10g/L H_2SO_4
Desorbed resin neutralisation target pH	5.0 – 6.0

Scandium Liquid Ion Exchange

Clean TeQ Sunrise has incorporated into the proposed flowsheet a continuous Liquid Exchange (cLX) circuit utilising Clean TeQ's proprietary Clean-IX® technology to selectively extract scandium from the nickel and cobalt eluate generated by the cRIP process.

Scandium adsorption loading onto the aminophosphonic (AP) chelating functional resin in a sulphuric acid matrix with pH of about 1-2 exceeds many other metals. The order of loading affinity on the AP ion exchange resin is:

Sc^{3+}, Fe^{3+}, Al^{3+}, Ti^{4+}, rare earth elements >> Ni^{2+}, Co^{2+}, Zn^{2+}, Fe^{2+}, Mn^{2+}, Ca^{2+}, Mg^{2+}

The nickel cobalt eluate is likely to have a residual free acidity of 10g/L H_2SO_4, indicating the pH range will be optimal for selective extraction of Sc^{3+} over the Ni^{2+} and Co^{2+} present in the nickel cobalt eluate. In addition, impurity trivalent cations such as Al^{3+}, Fe^{3+}, trace rare earth elements and Ti^{4+} in the nickel cobalt eluate will also be extracted onto the AP resin providing a means of impurity removal from the eluate.

A testwork program was conducted in the 2018 DFS to confirm and demonstrate the technology and application to the Project, and to develop and validate the flowsheet configuration and design criteria.

The testwork program was completed at both Clean TeQ's Melbourne laboratory and ALS Metallurgy and consisted of:

- assessment of alternative resins for the extraction of scandium from the nickel and cobalt eluate produced in the cRIP circuit
- desorption optimisation testwork completed as part of the 2016 scandium feasibility study
- adsorption loading cycle testwork incorporating the desorption processes to demonstrate impurity removal/scrubbing of the scandium loaded resin to reject impurities

- to make a low impurity scandium eluate to feed the scandium refinery.

The scandium cLX circuit included adsorption, desorption, resin scrub using dilute H_2SO_4 for the removal of base metal impurities, resin neutralisation to pH 7.5, elution using dilute Na_2CO_3 solution, resin regeneration back to the protonated (H^+) form with dilute H_2SO_4, and resin recycle back to adsorption.

Five resins were again assessed for the adsorption/desorption duty, and the Clean TeQ resin R603B was selected. Desorption testing confirmed that sodium carbonate at pH 10 was the best desorption solution.

Resin cycling through ten cycles of adsorption and desorption was carried out to evaluate the potential for impurity build up on the resin and to provide confirmation of design parameters. The desorption process performed as expected, with 99% of the scandium reporting to the combined eluate and desorption wash (used to make up the eluant to target Na_2CO_3 concentration).

The regeneration step demonstrated removal of residual impurities including sodium. The key design and performance criteria developed in the testwork are summarised in Table 13-5. The process will be piloted on eluate generated from the upcoming HPAL/cRIP pilot plant commencing in July 2018.

Table 13-5: Scandium cLX key process design criteria

Criteria	Measurement
Resin functional group	aminophosphonic (AP)
Number of adsorption stages (column)	1 duty, 1 standby
Scandium adsorption extraction	> 99%
Adsorption scrub solution	50g/L H_2SO_4
Scandium neutralisation pH	7.5
Eluant	150g/L Na_2CO_3
Scandium desorption extent	>99%
Desorption wash solution	reverse osmosis water
Desorption wash flowrate	1.5 BV/BVresin per hour
Scandium regeneration solution	50g/L H_2SO_4

13.3.6 Downstream Testwork at SGS Minerals, Lakefield, Canada

The downstream or ‘refining’ aspects of the processing flowsheet, i.e. following cRIP, were extensively tested at both batch and pilot plant scale from May to December 2017. The testwork included:

- Eluate Neutralisation (EN) to remove aluminium, chromium, iron, and most of the copper, from the nickel/cobalt eluate, by hydrolytic precipitation
- Impurity solvent extraction (ISX) to selectively remove zinc, calcium and manganese
- Cobalt solvent extraction (CSX) to selectively extract cobalt from magnesium and nickel, and produce a concentrated, high-purity cobalt sulphate liquor
- Nickel solvent extraction (NSX) to recover nickel and produce a concentrated, high-purity nickel sulphate liquor
- Batch cobalt sulphate liquor purification (CSP) to polish strip liquors free of trace impurities
- Cobalt sulphate crystallisation to produce hydrated cobalt sulphate crystals which met battery grade specification targets
- Batch nickel sulphate liquor purification (NSP) to polish strip liquors free of trace impurities

- Nickel sulphate crystallisation to produce hydrated nickel sulphate crystals which met battery grade specification targets
- Batch ammonium sulphate crystallisation (AXT).

These processes, using standard solvent extractants and typical industry conditions, have demonstrated and defined impurity removal extents as well as SX loading, scrubbing and stripping extents under specified conditions, and have generated process design criteria for the project. The tests were undertaken to support the 2018 DFS using both 1) synthetic eluate liquor and 2) real eluate derived from the 2016 Clean TeQ Sunrise HPAL/cRIP pilot plant. As a result, it is representative and mostly extensive enough to support the detailed design of the Clean TeQ Sunrise Project. The design parameters are also supported by industry practice for nickel and cobalt solvent extraction projects and operations which are well established.

However, some of the processes that were only tested on a relatively small scale in batch amounts may need further larger scale testing, using ore-derived or simulated liquors, in order to better define the process design criteria and demonstrate their suitability for the Clean TeQ Sunrise Project. In particular, these are the cobalt liquor purification processes of copper, zinc and manganese removal.

13.3.6.1 Pilot plant feed solutions

Two different feed compositions were used - solution simulating the project mass balance synthesised using high purity salts, and Clean TeQ Sunrise ore-based eluate produced in the 2016 PAL/cRIP pilot plant operation conducted at ALS Metallurgy, Perth. Throughout the piloting, the suffix A denoted synthetic eluate while suffix B denoted ore-based eluate. The block flow diagram depicted in Figure 13-8, summarises the order of test programs undertaken. All the downstream testwork was conducted at SGS Minerals Services, Lakefield Site, Canada (SGS Lakefield) during May-December 2017.

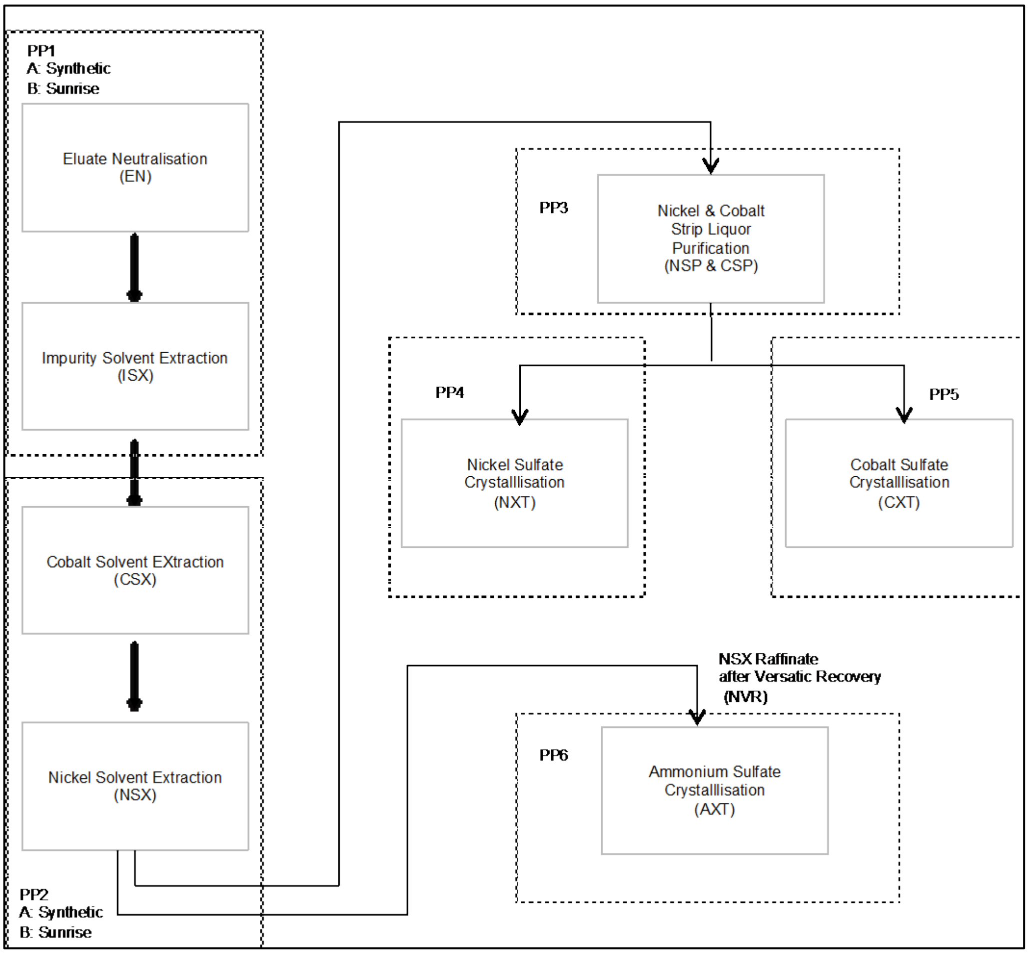


Figure 13-8: Downstream DFS Testwork Flow Diagram at SGS Lakefield

The synthetic eluate (13g/L Co and 40.6g/L Ni) was considered representative of the early years of commercial operation on a high-cobalt grade ore, whereas the ore-derived eluate (3.1g/L Co and 31.0g/L Ni) contained considerably less cobalt, especially relative to the nickel. The chemical compositions of the feed solutions are listed in Table 13-6. The ore-derived eluate contained significantly higher concentrations of most impurities, especially relative to the cobalt and nickel, and was ideal for testing and confirming the robustness of the flowsheet.

Table 13-6: Average composition of pilot plant feed solutions (as assayed)

PP1A Synthetic Solution (mg/L)				PP1B Ore-based Solution (mg/L)			
Co	13,381	Fe(II)	88	Co	3,149	Fe(II)	1,153
Ni	41,557	Mg	354	Ni	31,014	Mg	350
Al	,536	Mn	2,113	Al	2,953	Mn	718
Ca	568	Si	N.A..	Ca	668	Si	95
Cr	78	Na	52	Cr	182	Na	432
Cu	349	Zn	2,922	Cu	41	Zn	1,150
Fe(total)	148	H_2SO_4	10,000	Fe(total)	4,230	H_2SO_4	47,000

13.3.6.2 Eluate Neutralisation (EN)

The purpose of this circuit was to neutralise the residual acid and remove the iron, aluminium, chromium and most of the copper from the eluate prior to the impurity solvent extraction (ISX). This was achieved by treating the eluate with lime slurry in a series of five aerated reactors in series, to obtain a terminal pH of ~4.4. The pH was chosen to keep the nickel and cobalt co-precipitation to a minimum but maximise the rate of iron and aluminium removal. Air was sparged into the slurry, and the temperature controlled at 60ºC, to oxidise iron(II) to iron(III) to facilitate the iron precipitation. The final slurry was sent to a thickener from where precipitated solids were recycled back to the first EN reactor as seed.

These circuits are notoriously difficult to operate on a small scale, and the terminal pH was frequently higher than the target, often in the range 4.5 to 4.8, meaning nickel and cobalt precipitation was higher than was required for iron and aluminium removal. However, in PP1A, when the pilot plant was operating for an extended period under stable conditions, the concentrations of iron and aluminium could be reduced to <5mg/L and <1mg/L respectively, while the precipitation of nickel and cobalt were contained to 5%. In PP1B, co-precipitations of nickel and cobalt were significantly higher, at 21-26% and 6-9% respectively. This is believed to be due to the much higher concentrations of iron and aluminium, relative to nickel and cobalt, as well as the higher acidity of the feed, resulting in higher co-precipitation of nickel and cobalt by adsorption onto the precipitated iron and aluminium oxyhydroxides.

In the process design, the precipitated nickel and cobalt will be recycled back to the primary neutralisation feed solution for recovery of the metal values. Batch tests have shown that the precipitated nickel and cobalt, as well as the iron, aluminium and copper hydroxides, are completely dissolved in autoclave discharge liquors.

Based on the EN pilot plant results, utilising the results from PP1A which was more representative of the design feed liquor, the process design criteria in Table 13-7 were derived.

Table 13-7: Eluate Neutralisation Process Design Criteria

	Units	Value
$Ca(OH)_2$ concentration	%w/w	30
Operating temperature	ºC	60
Residence time	h	6
No of tanks	No	4
Air addition	Nm^3/m^3	4.2
Discharge slurry pH (60 C)	pH	4.4
Residual iron	mg/L	<5
Residual aluminium	mg/L	<1
Seeding	%	800
Flocculant	-	Magnafloc 336
Flocculant Addition	g/t solids	100
Specific settling area, design	$t/h/m^2$	0.10
Specific filtration rate	$kg/m^2/h$	300

13.3.6.3 Impurity Solvent Extraction (ISX)

In the pilot plant at Lakefield, the thickener overflow from the EN circuit reported directly to the ISX circuit. The purpose of the ISX circuit was to extract the impurity base metals (manganese, zinc and calcium) from the cobalt and nickel in the EN discharge solution. The organic extractant employed was di-2-ethylhexyl phosphoric acid (D2EHPA) in Exxsol D80 diluent, which is not totally selective for the impurity metals. The nickel and cobalt, together with magnesium and some copper, remained in the raffinate which advanced to the cobalt solvent extraction circuit (CSX).

Based on a series of batch tests, the ISX pilot plant circuit was configured consisting of 4 stages of extraction (pH 3.1 - 3.7) and 2 of scrubbing (pH 2.1 – 2.7) at 30°C, followed by 2 stages of stripping with hydrochloric acid, one stage of stripped organic wash, and one stage of raffinate washing with diluent, to minimize the transfer of entrained D2EHPA to the cobalt circuit. The pH in each stage was measured and controlled by addition of concentrated reagents: ammonium hydroxide in extraction, sulphuric acid in scrubbing, and hydrochloric acid in stripping. The raffinates from scrub stages were recycled to extraction. No attempt was made to re-use the raffinates from wash stages.

The pilot plant processed firstly synthetic (PP1A) and then actual Clean TeQ Sunrise ore-derived liquors (PP1B). The compositions of the feed solutions are listed in Table 13-8. The feed solution for PP1A contained 0.10moles/L of combined calcium, zinc and manganese, and a 22%v/v solution of D2EHPA was used for their extraction. In PP1B, the concentration of the impurities in the feed was 0.050 moles/L, and the extractant concentration was reduced to 10%. The molar ratio of Mn to Co in the PP1B feed (0.65:1) was almost four times higher than in PP1A (0.17:1), making the latter a more challenging separation. The PP1B circuit operated more smoothly, mainly due to the lessons learned in PP1A.

Table 13-8: Average composition of ISX feed solutions

PP1A Synthetic Solution (mg/L)				PP1B Ore-based Solution (mg/L)			
Co	13,359	Fe(total)	< 1	Co	2,363	Fe(total)	< 2
Ni	40,535	Mg	405	Ni	18,628	Mg	533
Al	< 3	Mn	2,133	Al	< 1	Mn	1,421
Ca	666	Na	54	Ca	555	Na	283
Cr	< 1	Zn	2,886	Cr	< 2	Zn	649
Cu	149			Cu	2		

The extraction and removal of the calcium, zinc and manganese was generally extensive throughout both the PP1A and PP1B campaigns. These metals were generally removed to <10mg/L Ca, < 2mg/L Zn, and < 2mg/L Mn. There was, as expected, co-extraction of some of the cobalt and magnesium, but these were very effectively displaced and scrubbed from the organic. The estimated average losses of cobalt to the strip product (waste) liquor were 0.6% and 0.8% respectively for PP1A and PP1B campaigns. The key process performance and design criteria developed in the pilot plant are listed in Table 13-9.

Table 13-9: Impurity Solvent Extraction Process Design Criteria

	Units	Value
Extractant	Name	D2EHPA
Diluent	Name	Exxol D80
Concentration Max	% v/v	20
Stoichiometric load	mol/mol%	38
Extraction stages	No.	4
Scrub stages	No.	2
Strip stages	No.	2
Strip org wash	No.	1
Diluent wash	No.	1
Temperature	ºC	30-60

13.3.6.4 Cobalt Solvent Extraction (CSX)

The duty of the cobalt solvent extraction (CSX) circuit was to fully extract cobalt from nickel and magnesium in the raffinate advancing from ISX. The organic extractant employed was bis(2,4,4–trimethylpentyl) phosphinic acid (Cyanex 272). With Cyanex 272, the cobalt to nickel separating factor in the pH range 5.0-5.5 is over 2,000 so any co-extracted nickel can be easily scrubbed off. The magnesium extraction can be minimised by efficiently scrubbing the loaded organic. The loaded organic was then stripped to produce cobalt-rich liquor for the cobalt purification circuit, where trace impurities of copper, zinc and manganese were to be removed ahead of crystallisation. Nickel remained in the CSX raffinate with magnesium and advanced to nickel solvent extraction (NSX).

Following a series of batch tests to define the configuration, a pilot plant circuit was set up consisting of four stages of extraction (Figure 13-9) and six stages of scrubbing, followed by two stages of stripping. Both synthetic and actual solutions were processed. The nickel-rich raffinate was washed with the diluent to remove any entrained extractant and added to the surge tank as the feed to nickel solvent extraction (NSX) which ran integrated with the CSX. The feed solutions to CSX for the pilot plant campaigns PP2A and PP2B are listed in Table 13-10.

Table 13-10: Average composition of CSX feed solutions

PP2A Synthetic Solution (mg/L)				PP2B Ore-based Solution (mg/L)			
Co	11,692	Fe(total)	<3	Co	1,966	Fe(total)	3.3
Ni	35,892	Mg	378	Ni	17,193	Mg	524
Al	<0.7	Mn	4	Al	<0.6	Mn	1.5
Ca	2	Na	49	Ca	1.7	Na	314
Cr	2	Zn	<9	Cr	<0.6	Zn	<3
Cu	5			Cu	0.7		

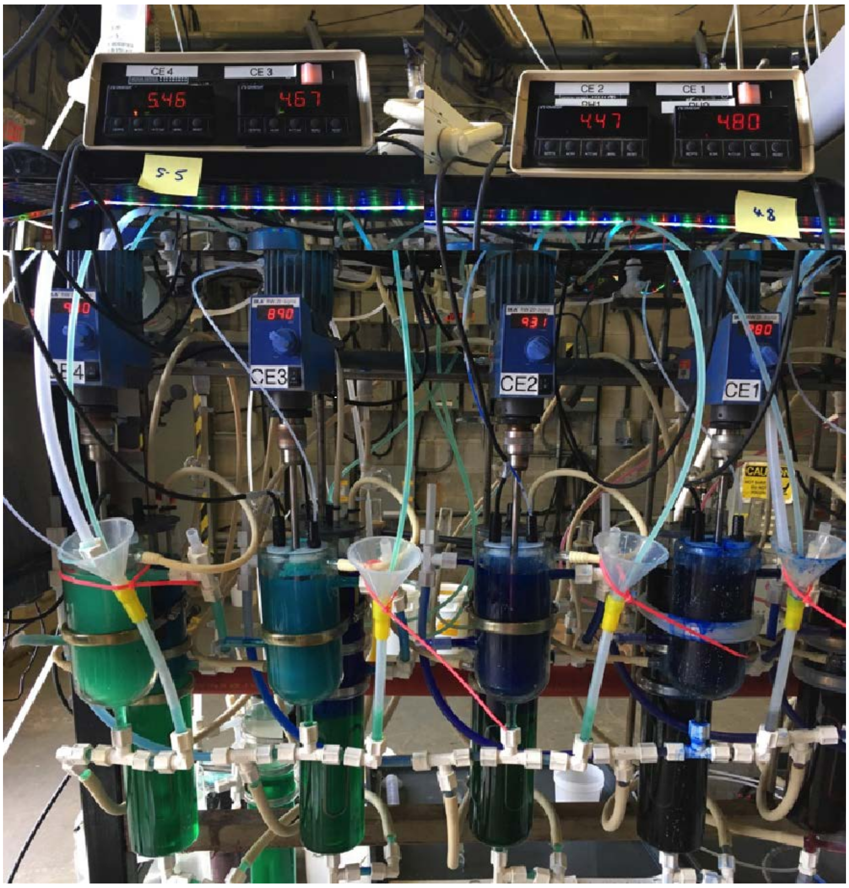


Figure 13-9: Cobalt solvent extraction – 4 stage extraction circuit.

- For PP2A, the circuit consisted of four stages of extraction, six stages of scrubbing, and two stages of stripping. Loaded and scrubbed organics were not washed. The nickel-rich raffinate was washed with the diluent to remove any entrained extractant and then passed through an anthracite/garnet coalescer to remove entrained organics. The raffinate was collected in pails and kept warm, analysed and added to the surge tank as the feed to PP2A nickel solvent extraction (NSX), which ran integrated with the CSX.
- For PP2B, with ore-based ISX raffinate as the feed, the circuit used was essentially the same; the only difference being the Cyanex 272 concentration, which was reduced from 20 to 11%v/v, diluted with Exxol D80. The pH in each stage was measured and controlled by addition of concentrated reagents - ammonium hydroxide in extraction, and sulphuric acid in scrubbing and stripping.
- Separation of the cobalt from the nickel was straightforward, but the extractability of the magnesium was similar to that of the cobalt and required more scrubbing stages than in the ISX circuit to ensure effective removal of co-extracted magnesium from the organic. In the scrub circuit, largely designed to remove the magnesium, its concentration in the organic had often already reached 1 or 2mg/L Mg by the fifth stage, so that in the final, low pH-stage, considerable cobalt was stripped, with little or no apparent removal of magnesium, in fact often reducing the Co to Mg ratio going forth to the strip stage.
- The Mg to Co molar ratio in the PP2B feed solution was almost four times higher than in PP2A; thus, making magnesium separation form the cobalt more difficult. As in PP2A, the magnesium in the organic in the scrub circuit organic had been reduced to between 1 and 2 mg/L by the third stage, and the low mixer pH of 2.9 in the final scrub resulted in considerable “stripping” of the cobalt, and again often resulted in a reduction in the Co: Mg ratio in the scrubbed organic. The strip liquor compositions are shown in Table 13-11.

Table 13-11: Typical composition of CSX strip solutions (average)

Synthetic Solution (mg/L)				Ore-based Solution (mg/L)			
Co	108,000	Fe(total)		Co	105,000	Fe(total)	81
Ni	10	Mg	1.5	Ni	3	Mg	2
Al		Mn	38	Al	21	Mn	64
Ca		Na		Ca	11	Na	314
Cu	45	Zn	5	Cu	29	Zn	32

- The aluminium, calcium and iron concentrations in this liquor were due to contamination of the feed solution from a garnet anthracite coalescer in one product pail during the early stages of circuit operation.
- The key process design criteria derived from the CSX pilot plant are presented in Table 13-12.

Table 13-12: Cobalt Solvent Extraction Process Design Criteria

	Units	Value
Extractant	Name	Cyanex 272
Diluent	Name	Exxol D80
Concentration Max	%v/v	30
Stoichiometric load	mol/mol %	50
Extraction stages	No.	4
Scrub stages	No.	6
Strip stages	No.	2
Diluent wash	No.	1
Temperature	C	50-60

13.3.6.5 Nickel Solvent Extraction (NSX)

The nickel solvent extraction circuit was designed to separate nickel from magnesium in the raffinate advancing from CSX. The organic extractant employed in NSX was Versatic Acid (V10). The NSX operating strategy was to fully extract nickel and direct magnesium to raffinate together with ammonium (from pH control in earlier solvent extraction processes). The loaded organic would then be stripped to produce nickel-rich liquor for nickel sulphate recovery by crystallisation. The raffinate advanced to ammonium sulphate crystallisation. The NSX and the CSX circuits were run integrated during the piloting program.

Following a series of batch tests to define the configuration, a pilot plant circuit was set up to process the synthetic feed, consisting of five extraction stages plus a V10 recovery stage, three scrub (initially two), two strip stages and an acidification stage for the raffinate, to enhance the recovery of dissolved organic extractant (Figure 13-10).

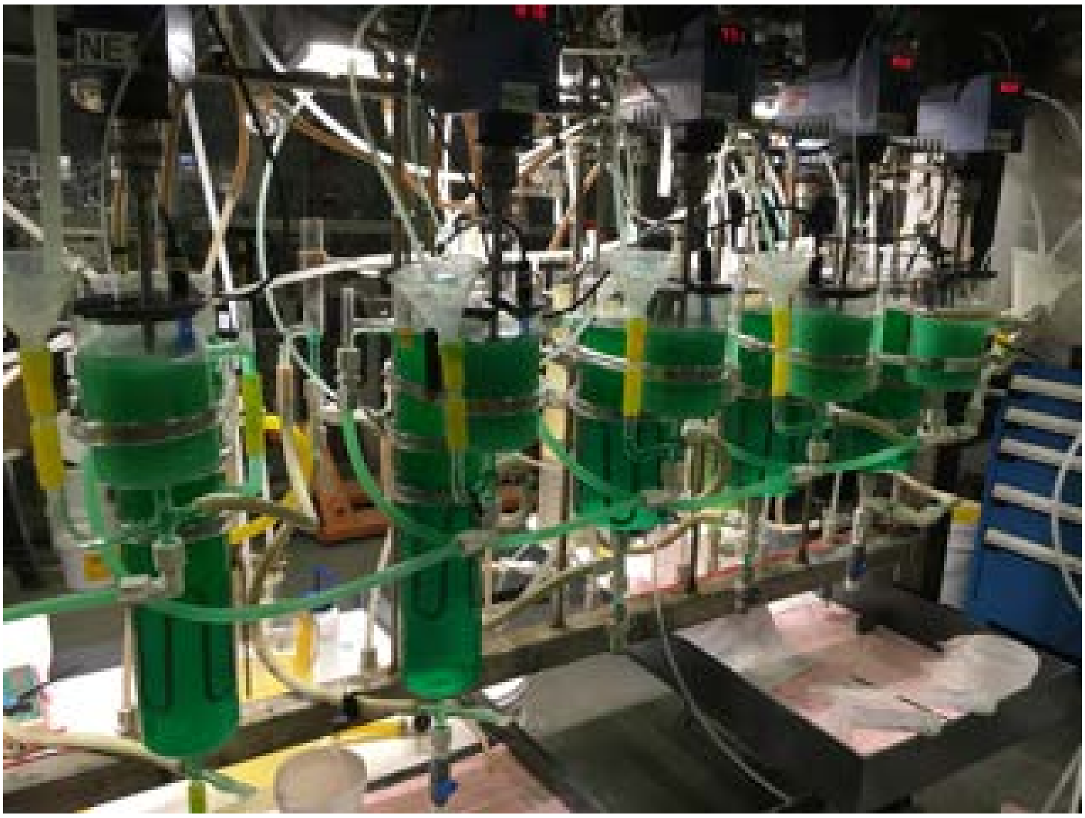


Figure 13-10: Nickel solvent extraction – 5-stage extraction circuit

The extractant concentration in the Exxsol D80 diluent was 58% for PP2A. This was reduced to 29% for PP2B (due to the total moles of nickel to be loaded being half that in PP2A). It was noted that the elevated extractant concentration used in PP2A appeared very viscous so, as an adjustment after the pilot, the decision was made to reduce the V10 concentration to 30% in the PDC. Other changes in PP2B were the reduction to 4 stages in extraction, while the scrub was increased to four stages, and the strip to three.

The feed to the circuit (Table 13-13), as individual pails of CSX raffinate, varied considerably in nickel concentration, from 27 to 38g/L Ni (median 33.5). It varied more radically in magnesium concentrations, from 1 to 673mg/L, as a result of upsets in the cobalt SX circuit. In spite of these variations, the nickel SX circuit achieved and maintained a very high extraction of the nickel, with the raffinate containing a median of 40mg/L Ni (99.9% recovery), with excellent rejection of the magnesium to the raffinate.

Table 13-13: Composition of NSX feed solutions (average)

A. Synthetic Solution (mg/L)				**B. Ore-based Solution (mg/L)**			
Co	< 2	Fe(total)	< 1	Co	1.36	Fe(total)	< 2
Ni	33,522	Mg	295	Ni	15,533	Mg	464
Al	< 0.3	Mn	< 0.05	Al	< 3	Mn	< 0.4
Ca	2.5	Na	29	Ca	2.3	Na	290
Cr	< 1	Zn	< 3	Cr	< 1	Zn	< 7
Cu	< 0.4			Cu	< 1		

The median strip product solution composition was 109g/L Ni, with only 3mg/L Co, and 3mg/L Mg, but also contained 4mg/L zinc which required subsequent removal by ion exchange resin. The nickel circuit raffinate, as feed to ammonium sulphate recovery, contained 30mg/L Ni, 270mg/L Mg and about 130g/L $(NH_4)_2SO_4$.

Extraction of the nickel in the PP2B campaign was also highly efficient. The product strip liquor averaged 103g/L nickel (99.8% recovery), 3mg/L magnesium and 4.5mg/L cobalt. The product exceeded the battery grade specification targets of Ni: Mg and Ni: Co of 22,000 and 4,400, respectively. The strip product also contained 12mg/L zinc, which was unexpected, as zinc was below detection limit in all the aqueous (<2mg/L) and organic (<7mg/L) streams analysed in NSX circuit except in the strip liquor. The specification target for Ni: Zn mass ratio is 44,000, hence the strip product required subsequent removal by ion exchange resin ahead of crystallisation.

The process design criteria derived from the NSX pilot plant are presented in Table 13-14.

Table 13-14: Nickel Solvent Extraction Process Design Criteria

	Units	Value
Extractant	Name	Versatic 10
Diluent	Name	Exxol D80
Concentration Max	%v/v	30
Stoichiometric load	mol/mol%	38
Extraction stages	No.	5
Scrub stages	No.	3
Strip stages	No.	2
Organic recovery	No.	1
Diluent wash	No.	1
Temperature	ºC	50-60

An important feature of these campaigns was the gradual build-up of phosphorus in the Versatic extractant solution, presumably due to the carryover of Cyanex 272 from the cobalt SX circuit. Such carryovers, although not important in this context, stress the need at full scale for adequate and effective removal of organics from raffinates passing from one type of extractant to another, especially with three different reagents in the overall process.

13.3.6.6 Zinc removal from nickel sulphate strip liquor (PP3)

The nickel sulphate strip liquor from the nickel solvent extraction (NSX) circuit advanced to the nickel sulphate crystallisation. The solution was expected to be suitable as the feed to crystallisation circuit without any further purification as any zinc should have been removed in the impurity removal stage (ISX). However, the solution produced from the NSX continuous pilot plant contained trace amount of zinc (possibly from sample contamination) which needed to be removed prior to crystallisation. An ion exchange resin VP OC 1026 was used to remove the zinc from the nickel strip liquor following pH adjustment to about 2.1, ahead of crystallisation. The zinc content of the liquor was successfully reduced from 10mg/L to <0.02mg/L.

Zinc removal from the nickel crystallisation feed liquor will not initially form part of the Clean TeQ Sunrise Project flowsheet, although space will be allocated to include the process if required. An assessment of the zinc issue has determined it was most likely due to sample contamination. Further testing is planned post-DFS, and definitely recommended, to further investigate whether the zinc content of the nickel crystallisation liquor was real or due to sample contamination.

13.3.6.7 Nickel sulphate crystallisation pilot plant (PP4)

A portion of the strip liquors generated from the continuous nickel solvent extraction (NSX) pilot plant, both synthetic and ore-based, after treatment to remove zinc, were fed to a nickel sulphate crystallisation pilot plant (NXT), which operated continuously for 117 hours.

The composition of the feed liquors to the pilot plant are shown in Table 13-15.

Table 13-15: Composition of NXT feed solutions (average)

PP4A Synthetic Solution (mg/L)				**PP4B Ore-based Solution (mg/L)**			
Co	0.72	Fe(total)	<0.2	Co	4.45	Fe(total)	<0.2
Ni	106,000	Mg	1.74	Ni	104,000	Mg	0.21
Al	<0.2	Mn	0.04	Al	<0.2	Mn	<0.04
Ca	<0.9	Na	7	Ca	<0.9	Na	15
Cr	<0.2	Zn	<0.02	Cr	<0.1	Zn	<0.1
Cu	<0.1	P	<5	Cu	<0.08	P	<5

A total of 14.6kg of hydrated nickel sulphate was produced in 36 batches of which 9.4kg was from ore-based liquor. The assays of typical samples of nickel sulphate produced from the ore-based liquor are summarised in Table 13-16.

Table 13-16: Composition of hydrated nickel sulphate crystals

From Synthetic Solution (g/t)				**From Ore-based Solution (g/t)**			
Batch 5		Batch 10		Batch 25		Batch 30	
Co	3.0	Co	4.0	Co	6.7	Co	7.1
Ni	220,000	Ni	225,000	Ni	222,000	Ni	223,000
Mg	2.0	Mg	1.6	Mg	<0.8	Mg	<0.7
Na	<5	Na	<5	Na	6	Na	7
Zn	<20	Zn	<20	Zn	<1	Zn	<1
Cu	<1	Cu	<1	Cu	<1	Cu	<1
H_2O	6.4	H_2O	nd	H_2O	6.4	H_2O	5.8

H_2O = hydration number (moles water/mole $NiSO_4$) nd= not determined.

The only elements that analysed above the detection limit in crystals were nickel, cobalt, magnesium and sodium. Other elements analysed at below detection, but not listed above, included Al (<2), Ca (<5), Cd (<2), Cr (<2), Fe (<2), Mn (<0.4), Pb (<0.2), and Si (<7 g/t).

The concentration of cobalt averaged 3.5g/t for synthetic based and 7.1g/t for ore-based crystals. These are well below the specification target of 50g/t cobalt. Magnesium averaged 2.1 and 1.2g/t for synthetic and ore-based respectively. Sodium content of the products averaged 6.8g/t, again well below the specification of 20g/t.

The average nickel and total sulphur content of the ore-based crystals were 22%w/w and 11.9%w/w, respectively. The average hydration number calculated based on the sulphur content was 6.4. Table 13-17 shows the average values of SGS analytical data on nickel sulphate hexahydrate produced from ore-based liquor against the battery market specification targets.

Table 13-17: Assay of Sunrise $NiSO_4.7H_2O$ Against Specification Target

Element	$NiSO_4.7H_2O$ (g/t)	
	Target Specification	Product
Aluminium (ppm)	< 5	<2
Calcium (ppm)	< 10	<5
Cadmium (ppm)	< 5	<2
Cobalt (ppm)	< 50	7.1
Chromium (ppm)	< 5	<2
Copper (ppm)	< 5	<1
Iron (ppm)	< 5	<2
Magnesium (ppm)	< 10	1.2
Manganese (ppm)	< 5	<0.4
Sodium (ppm)	< 20	6.8
Nickel (%w/w)	> 22.00%	22.06%
Lead (ppm)	< 5	<0.2
Silicon (ppm)	< 10	<7
Zinc (ppm)	< 5	<1

The nickel sulphate hexahydrate produced from both the synthetic and ore-based solution met the specification targets. Assays at two independent laboratories confirmed the SGS analyses. The continuous crystallisation testwork has shown that high purity nickel sulphate crystals can be produced from the Clean TeQ Sunrise ore. A photograph of the packed samples sent for market analysis is shown in Figure 13-11.

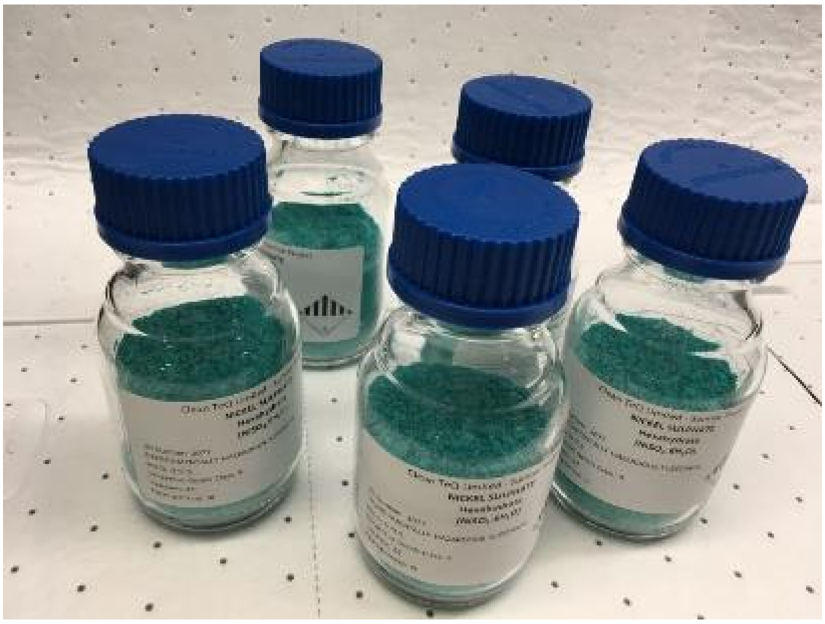


Figure 13-11: Packed Samples of Sunrise $NiSO_4.6H_2O$ Sent for Market Analysis

13.3.6.8 Cobalt liquor purification (PP3)

Prior to crystallisation, in order to meet the required specification targets, the cobalt liquor from the continuous cobalt solvent extraction (CSX) pilot plant operation needed to be further purified by removal of copper, zinc and manganese. The assays of the feed liquor for cobalt purification, for both synthetic and ore-based eluate, are shown in Table 13-18.

Table 13-18: Analysis of Cobalt Sulphate Strip Liquor from CSX 2A and 2B Pilot Plants

Solution	Solution Assay (mg/L)										Free Acid (g/L)
	Co	Ni	Cu	Zn	Mn	Fe	Al	Ca	Mg	Ti	
PP2A	91,900	2	20.6	7.8	37.8	3.0	0.6	<0.9	0.15	<0.2	13.3
PP2B	86,000	3	29.7	26*	58.9	75.1*	20.3*	12.0*	1.57	9.38	38.8

* Due to prior contamination of the sample.

Testwork at Veolia Water Solutions & Technologies laboratory in Plainfield, IL, USA, had indicated that, if not removed prior to crystallisation, these impurities will report to the crystal, making the cobalt sulphate product off-specification. Therefore, several purification stages were developed to remove copper, zinc and manganese from this liquor.

The unusually high iron, aluminium, calcium and zinc values in PP2B liquor were due to a contamination (discussed previously). This solution was also subjected to the purification regime and provided a good test of the robustness of the cobalt purification flowsheet.

The solution from CSX contained excess acid, and the pH had to be raised prior to the use of IX resins. To avoid the introduction of alkali cations such as sodium, the preferred neutralising agent for this duty was cobalt carbonate prepared from the cobalt strip liquor and high purity sodium carbonate.

Following a batch testwork program to assess suitable ion exchange resins and manganese removal options, the following testing program was implemented for bulk cobalt liquor purification:

- production of cobalt carbonate for neutralisation duty to increase of pH of cobalt strip liquor to pH 5.0 prior to ion exchange (IX)
- copper removal using LANXESS TP207 IX ((imino-diacetic acid) resin.
- zinc removal using LANXESS VP OC 1026 (D2EHPA impregnated) resin
- manganese(II) removal by oxidation/ precipitation as MnO_2
- continuous cobalt sulphate crystallisation pilot plant operation.

Copper Removal

Copper removal from cobalt strip liquor was performed using LANXESS TP207 (imino-diacetic acid resin) at pH 5 (although pH as low as 1.4 were effective) and ambient temperature. The large volume (about 13L) batch test data for the copper loading test, giving both feed and eluate analyses, are given in Table 13-19. Copper in the liquor was reduced from 8.9mg/L in the feed to 0.4mg/L in the effluent. As can be seen from the assays, the cobalt liquor feed used in this test had already been through the manganese removal step, but not zinc removal, as is evident from assay.

Table 13-19: Scavenging of Copper Using TP207 Resin – Production Run (PP3)

Solution ID	pH	Solution Assay (mg/L)		
		Copper	Zinc	Manganese
Feed	5.0	8.9	38.4	<0.9
Effluent	1.68	0.40	36.3	<0.9

Flowrate:2.0 BV per hour, down flow; total copper loading: 110 mg/L_{WSR} or 3.5 meq/L_{WSR}.

Two step elution tests at 20 and then 150g/L H_2SO_4 showed that cobalt was eluted by both the weak and strong acid while copper was only eluted by the strong acid. The elution was not optimised.

Zinc Removal

Polishing zinc extraction from cobalt strip liquor was performed using LANXESS VP OC 1026 (D2EHPA impregnated) resin at pH 3 and ambient temperature. The loading test feed and eluate data are given in Table 13-20. The cobalt liquor feed used in this test had already been through the manganese removal stage, but not copper removal, as is evident from assay.

Table 13-20: Zinc loading data for VP OC 1026 (D2EHPA) IX Resin

Solution ID	Solution Assay (mg/L)			
	Zinc	Manganese	Copper	Aluminium
Feed	6.7	0.82	16	0.9
Eluate	0.02	0.66	15	0.2

The resin was very effective in scavenging the zinc, reducing the concentration from 6.7 mg/L in the feed to 0.02 mg/L in the effluent. In addition, the resin also extracted the aluminium (and in other tests iron, calcium and titanium), while copper was not touched. These tests showed that the VP OC 1026 can clean up potential contamination of cobalt strip liquor from several other elements as well as zinc.

- Elution of zinc extraction resin that had been loaded from a cobalt solution was carried out with two strengths of acid, 10g/L H_2SO_4 followed by 50g/L H_2SO_4. Zinc and cobalt were largely eluted by the weak acid, as was manganese. The loaded resin contained a small amount of aluminium and iron which was only eluted by the stronger acid.
- Combined Copper and Zinc Removal

To simplify the cobalt purification flowsheet, a test was designed and conducted where two different IX resins in columns, TP207 followed by VP OC 1026, were connected in series, without pH adjustment in between. The TP 207 column was pre-loaded with cobalt by pumping 1.7 L of feed liquor (effluent pH 1.7) before allowing the effluent to pass on to the VP OC 1026 column. A total of 20 BV was processed. The feed contained 13.6 mg/L copper and 9.0 mg/L zinc. Copper was reduced to <0.2 mg/L in the TP 207 effluent while the zinc was below <0.5 mg/L in the VP OC 1026 effluent.

- Manganese removal

Oxidative precipitation of manganese as manganese dioxide from cobalt-rich liquor using Caro's acid (peroxymonosulphuric acid, H_2SO_5) was selected as the best option. Caro's acid is prepared by the reaction between hydrogen peroxide and sulphuric acid. The reaction between Caro's acid and manganese(II) ions to produce manganese dioxide can be represented as:

$$Mn^{2+} + H_2SO_5 + H_2O \rightarrow MnO_2 + H_2SO_4 + 2H^+$$

- The results showed that, using 500% Caro's acid at 50°C, the manganese could be reduced to below 1mg/L in cobalt rich liquors with cobalt losses less than 0.1%.
- The testwork for copper, zinc and manganese removal was only completed in batch tests as described above. Although these tests showed proof of the concept, and the ability of the flowsheet to achieve the desired result, more testwork is required to optimise the flowsheet parameters and demonstrate these process steps on a larger scale.

13.3.6.9 Cobalt sulphate crystallisation pilot plant (PP5)

A bulk sample of cobalt sulphate heptahydrate ($CoSO_4.7H_2O$), which met the required product specification target, was produced in a continuous cobalt sulphate crystallisation pilot plant (CXT) campaign conducted at SGS, Lakefield in November 2017. The feed for this circuit was about 25L of cobalt strip liquor from the continuous solvent extraction pilot plant, which had then been through the impurity removal batch test program to remove copper, zinc and manganese contaminants. The pH was then adjusted to >3 using high purity cobalt carbonate. The pilot plant operated continuously for ~90 hours over 4 days and processed 11.5 litres of synthetic and 13.1 litres of ore-based feed solution. A total of 8.7kg of cobalt sulphate was produced in 21 batches. The average moisture was 0.1%w/w.

Prior to this pilot plant campaign, a test program was conducted at Veolia to determine the effect of feed quality on the cobalt sulphate crystal purity. The results showed that copper, manganese, magnesium, nickel and zinc reported to the crystal in significant quantities, while sodium, aluminium, and silicon did not report to the crystal at detectable levels, but rather concentrated in the mother liquor. The composition of the feed liquors to the pilot plant are shown in Table 13-21.

Table 13-21: Composition of CXT feed solutions (average)

PP5A - Synthetic Solution (mg/L)				**PP5B - Ore-based Solution (mg/L)**			
Co (g/L)	104	Fe(total)	<2	Co (g/L)	105	Fe(total)	<2
Ni	2.3	Mg	0.48	Ni	3.2	Mg	2.22
Al	3	Mn	1.05	Al	1.6	Mn	<0.08
Ca	2	Na	2	Ca	1.7	Na	6
Cr	<0.2	Zn	<0.5	Cr	<0.2	Zn	<0.9
Cu	0.4	P	<5	Cu	0.4	P	<5

- Typical assays of representative samples from four of the 21 batches of cobalt sulphate crystals produced during the continuous crystallisation pilot plant are summarised in Table 13-22.

Table 13-22: Composition of hydrated cobalt sulphate crystals

PP5A - From Synthetic Solution (g/t)				**PP5B - From Ore-based Solution (g/t)**			
Batch 2		**Batch 6**		**Batch 12**		**Batch 20**	
Co	216,000	Co	223,000	Co	211,000	Co	238,000
Ni	<6	Ni	<6	Ni	<7	Ni	<7
Mg	<0.7	Mg	<0.7	Mg	2.6	Mg	3.5
Mn	<0.4	Mn	<0.4	Mn	<0.4	Mn	<0.4
Zn	<1	Zn	<1	Zn	<2	Zn	<2
Cu	<1	Cu	<1	Cu	<1	Cu	<1
H_2O	7.4	H_2O	7.1	H_2O	6.5	H_2O	7.3

H_2O = hydration number (mole/mol). Batches 1-6 synthetic, 7-10 transitional; 11-21 ore-based.

Apart from sulphur and water, the only elements above detection limits in crystals were cobalt and magnesium. The concentration of magnesium in synthetic and ore-based feeds were 0.48 and 2.22 mg/L, respectively. The average cobalt and total sulphur contents of the ore-based crystals were 21.6%w/w and 11.3%w/w respectively. The average hydration number (calculated based on the sulphur content) was 7.2. Magnesium averaged 2.8g/t, well below the specification target of 10g/t.

Table 13-23 shows the average values of SGS analytical data on cobalt sulphate heptahydrate produced from ore against the specification targets.

Table 13-23: Assay of Sunrise $CoSO_4.7H_2O$ Against Specification Target

Element	$CoSO_4.7H_2O$ Target	$CoSO_4.7H_2O$ Product
Aluminium (ppm)	< 5	<2
Calcium (ppm)	< 20	<5
Cadmium (ppm)	< 5	<3
Cobalt (%w/w)	> 20.80%	21.60%
Chromium (ppm)	< 5	<1
Copper (ppm)	< 10	<1
Iron (ppm)	< 5	<5
Magnesium (ppm)	< 10	2.8
Manganese (ppm)	< 5	<0.4
Sodium (ppm)	< 20	<5
Nickel (ppm)	< 50	<7
Lead (ppm)	< 5	<0.2
Silicon (ppm)	< 10	<7
Zinc (ppm)	< 5	<2

Cobalt sulphate heptahydrate produced from both the synthetic solution and ore-based solution met the specification targets for the battery market. Assays at two independent laboratories confirmed the results. A photograph of the packed samples sent for market analysis is shown in Figure 13-12.

Figure 13-12: Packed Samples of Sunrise $CoSO_4.7H_2O$ Sent for Market Analysis

13.3.6.10 Ammonium sulphate crystallisation (PP6)

Two batch ammonium sulphate evaporative crystallisation tests (each with 4L of feed solution) were performed using composite raffinate samples from the nickel solvent extraction circuit. The compositions of two composites used are given in Table 13-24.

Table 13-24: Analysis of nickel raffinate feed composites

Solution ID	Solution Assay (mg/L)							Solution Density (g/mL)
	$N_{(Total)}$ g/L	$S_{(Total)}$ g/L	Ni	Mg	Ca	Na	Cl	
PP6A Feed	27.9	30.3	30.1	272	1.56	46	55	1.068
PP6B Feed	11.5	14.0	27.8	447	1.72	273	86	1.030

The tests produced 615 g and 234 g respectively of ammonium sulphate crystals. Assays of the products are given in Table 13-25 and a photo of the product in Figure 13-13.

Table 13-25: Analysis of Ammonium Sulphate Products from Batch Tests

$(NH_4)_2SO_4$ Product ID	Solids Assay (g/t)						
	$N_{(Total)}$ %	$S_{(Total)}$ %	Ni	Mg	Ca	Na	Cl
PP6A	21.1	22.4	223	1910	<9	268	118
PP6B	21.3	23.0	494	7530	20	4660	375

Both products contained total nitrogen and sulphur at 21% and 22-23% respectively, resembling the theoretical values of ammonium sulphate. Some nickel (0.02-0.05%) and magnesium (0.2-0.75%) were also present. Sodium and calcium concentrations increased significantly in the mother liquor but not the product (Table 13-26).

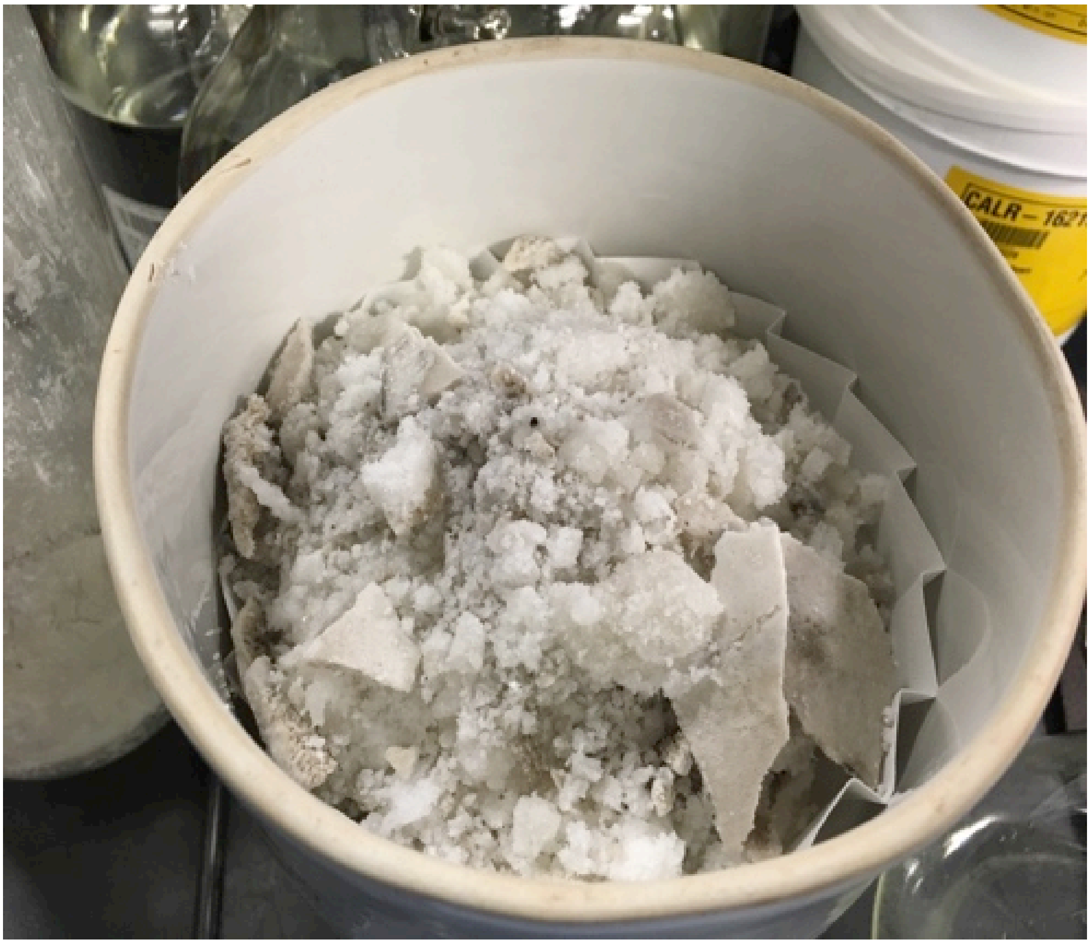


Figure 13-13: Ammonium sulphate (Amsul) product

Table 13-26: Analysis of Mother Liquor After Crystallisation (PP6A)

$(NH_4)_2SO_4$ Product ID	Solids Assay (g/t)						
	$NH_{4(Total)}$ %	$SO_{4(Total)}$ %	Ni	Mg	Ca	Na	Cl
PP6A Mother Liquor	10.8	11.8	66	1,402	24	654	700

The product specification for the Amsul are shown in Table 13-27.

Table 13-27: Amsul product specification

	Unit	Value
Product Size. Particle Size Distribution, P_{10}	mm	2
Nitrogen – ammoniacal (minimum)	% w/w	21.0
Sulphur – water soluble (minimum)	% w/w	24.0
Moisture (maximum)	% w/w	< 0.5
Insoluble fraction	% w/w	0.05
Free acid, H_2SO_4 (maximum)	% w/w	0.05
Cadmium, Cd	ppm	< 10
Lead, Pb	ppm	< 500
Mercury, Hg	ppm	< 5

13.3.7 Future testwork

At the time of completion of the Definitive Feasibility Study (June 2018) and the completion of this report (July 2018), there remain several testwork programs in progress. These programs have been initiated by Clean TeQ to optimise the process routes and further reduce any technical risk, provide further data for engineering design and provide further information for process development.

13.3.7.1 Current testwork programs

An overview of testwork programs currently in progress and due for completion in October 2018 are:

- Ore mineralogy techniques to better determine ore mineralogy on site
- Ore beneficiation upgrade potential via low charge attrition milling of the ore
- Potential for both gravity rejection of chromite and gravity recovery of platinum
- HPAL/cRIP pilot plant evaluating alternative commercially available resin plus associated batch testwork to support plant design
- Thickening testwork on samples generated from pilot operation from HPAL feed thickener duty to neutralised tailings duty
- Rheology mapping of slurry process streams from HPAL feed through HPAL discharge, PN slurry, cRIP slurry and both acidified and neutralised tailings.

13.3.7.2 Future testwork programs

After completion of these activities, Clean TeQ are planning a future testwork program to provide greater confidence in expected performance of the ore from ramp-up of operations through to the initial 5 to 10 years of operation for the Project. Relevant programs included in this phase of work are:

- Fully comprehensive geometallurgical modelling of the initial 5-10 years of ore to be treated including mineralogy, ore beneficiation, HPAL feed slurry settling and rheology, HPAL leach extraction and chemistry, and likely implications on cRIP and refinery performance to produce high specification sulphate products
- Development of metallurgical unit operation models to assist the Project in initial ramp up planning through first ore to name plate capacity. This is expected to assist commissioning and operations in the diagnosis of process issues and provide resolution strategies
- Zinc, copper and manganese removal from cobalt liquor flowsheet optimisation
- Scandium refinery flowsheet optimisation / value enhancement
- Opportunities for further value metal recovery from the Project.

14 Mineral Resource Estimates

14.1 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 55,611m. 1,354 holes were Reverse Circulation (RC), with the remainder (148) being air core (AC) holes (see Figure 14-1).

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.

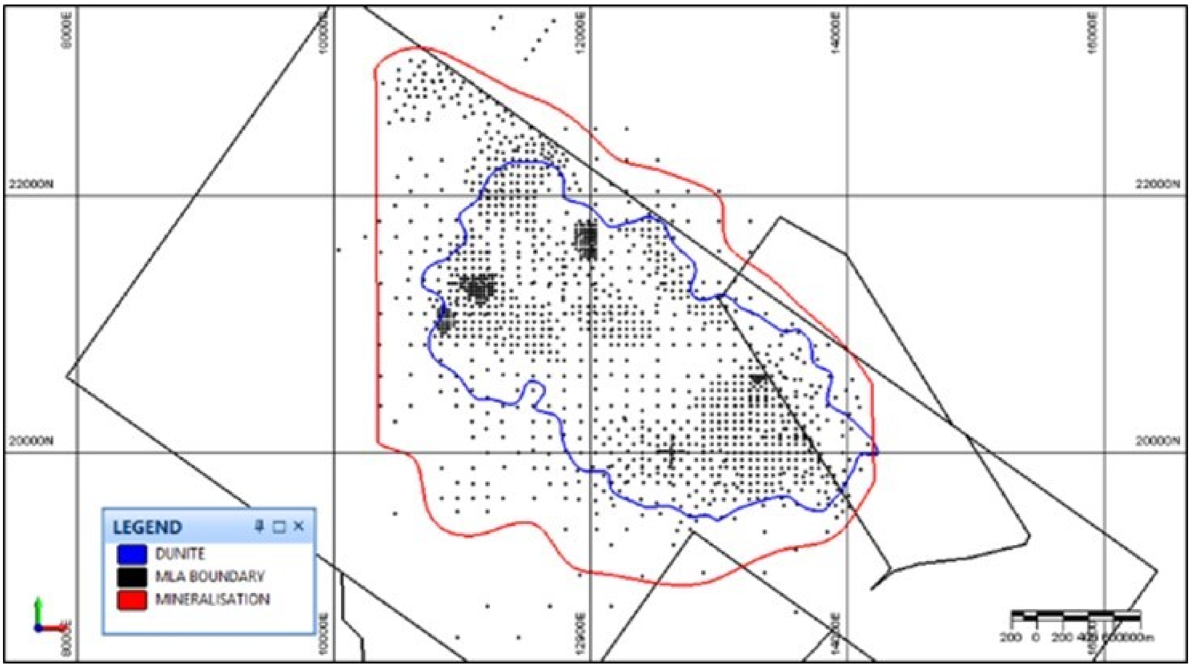


Figure 14-1: Drill Hole Location Plan

14.1.1 Sample Lengths and Compositing

A histogram of raw assay lengths is shown in Figure 14-2. On the basis that the majority of samples are 1m in length, it was decided to composite the assay data to 1 m. All composites had a length of 1 m.

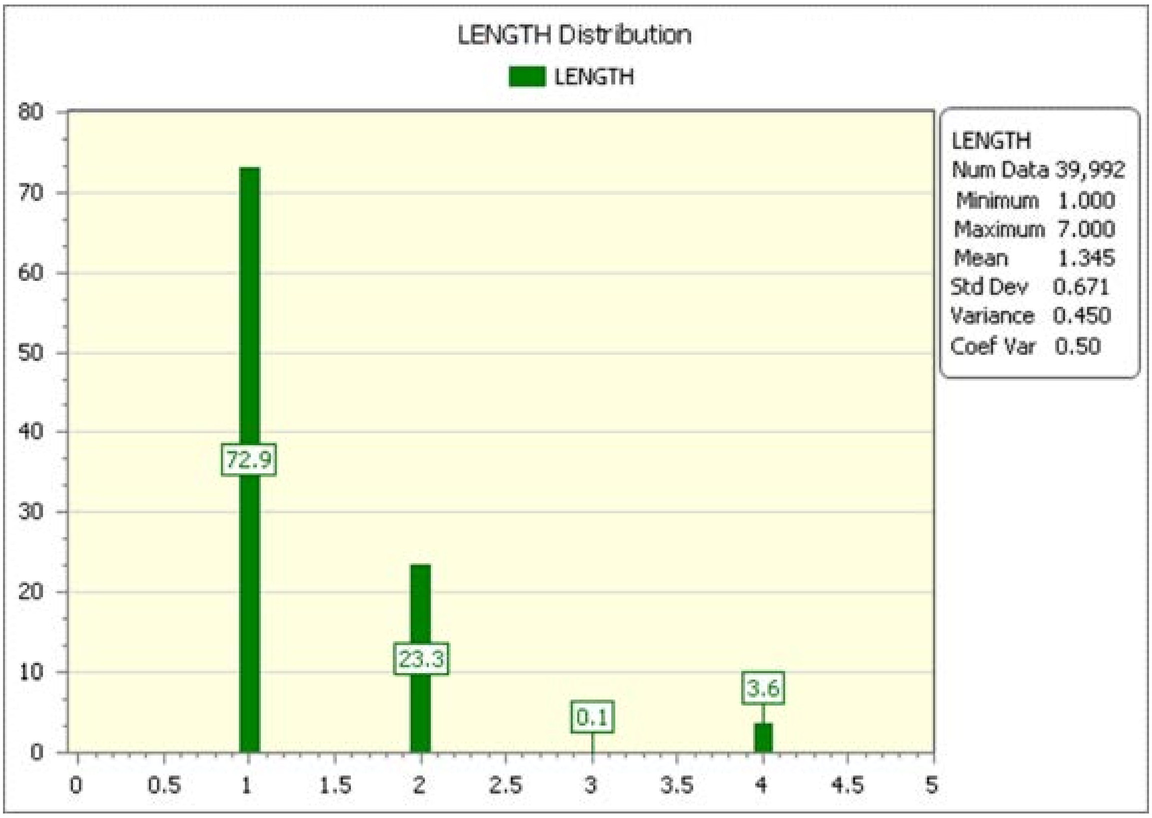


Figure 14-2: Assay Sample Length Histogram

14.1.2 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 shown in Table 14-1 to Table 14-3.

Table 14-1: Summary Statistics of Major Elements For Composites – Normal Statistics

Statistic	Nickel (%)	Cobalt (%)	Scandium (ppm)	Platinum (ppm)	Iron (%)	Aluminium (%)	Silicon (%)	Manganese (%)
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
Standard deviation	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
Standard error	0.002	0	0.484	0.002	0.071	0.014	0.06	0.019
Coefficient variation	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

Table 14-2: Summary Statistics of Major Elements For Composites – Data

Data Criteria	Element							
	Nickel	Cobalt	Scandium	Platinum	Iron	Aluminium	Silicon	Magnesium
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

Table 14-3: Summary Statistics of Major Elements for Composites – Percentiles

Percentile	Element							
	Nickel	Cobalt	Scandium	Platinum	Iron	Aluminium	Silicon	Magnesium
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

14.1.3 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. The distribution of cobalt, nickel and scandium is clearly very different in each of the domains as shown in Figure 14-3 to Figure 14-5.

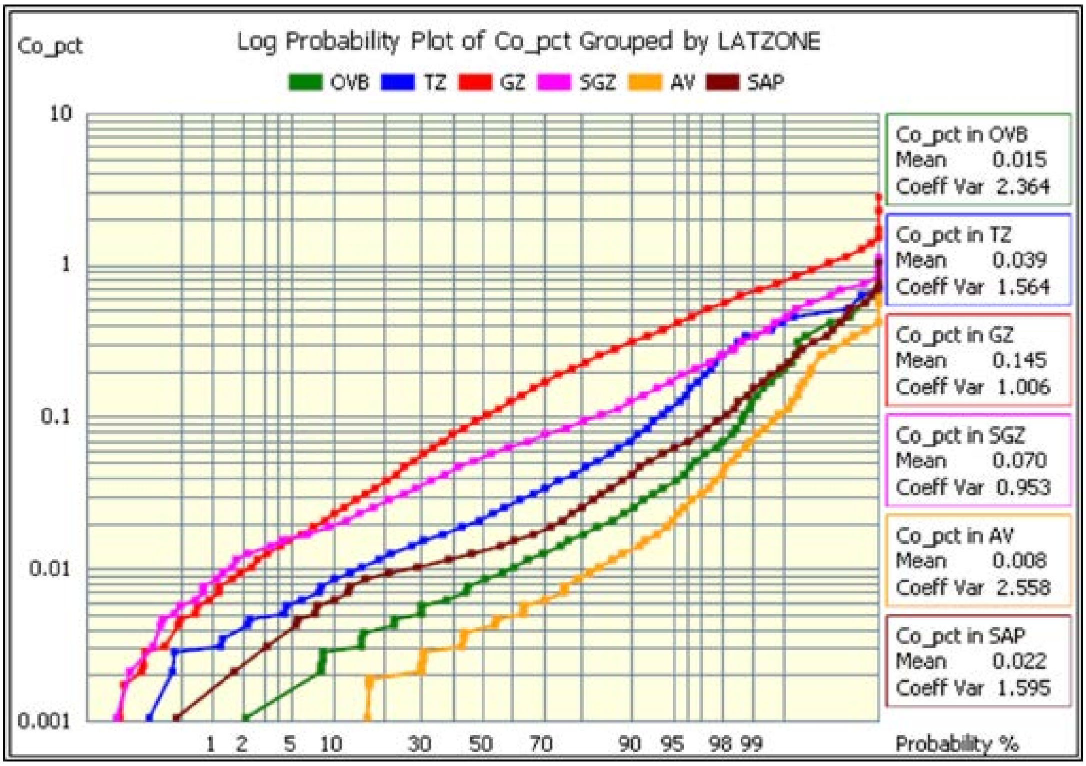


Figure 14-3: Cobalt Distribution

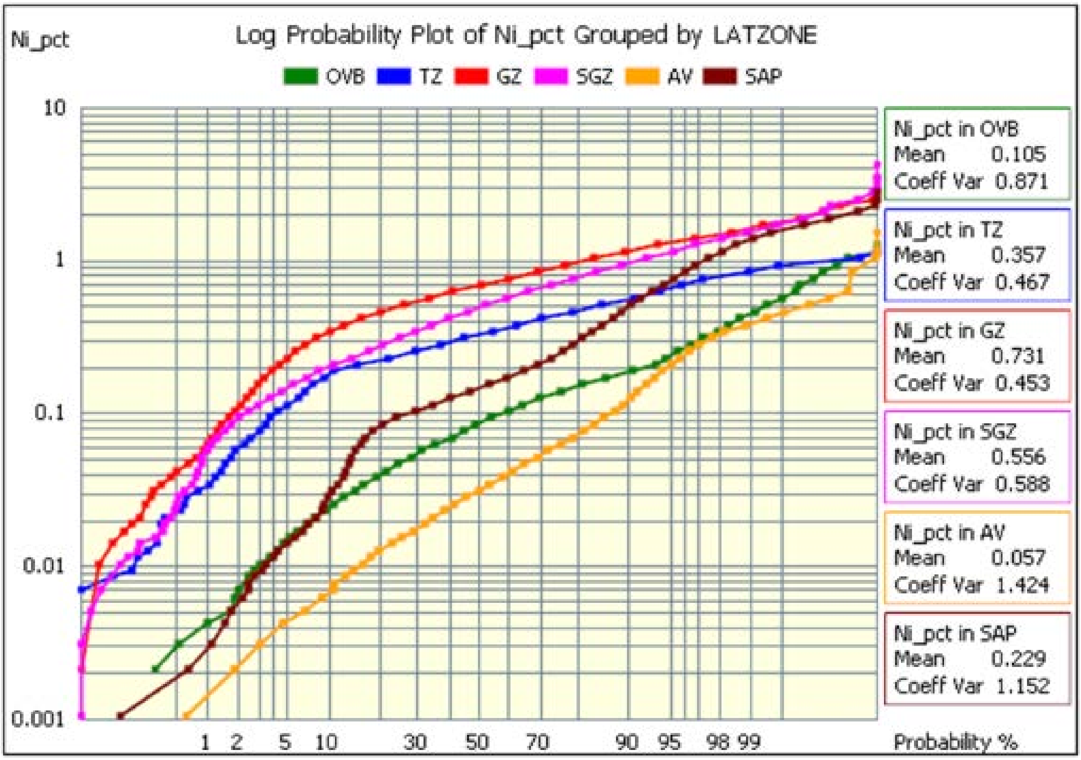


Figure 14-4: Ni Distribution

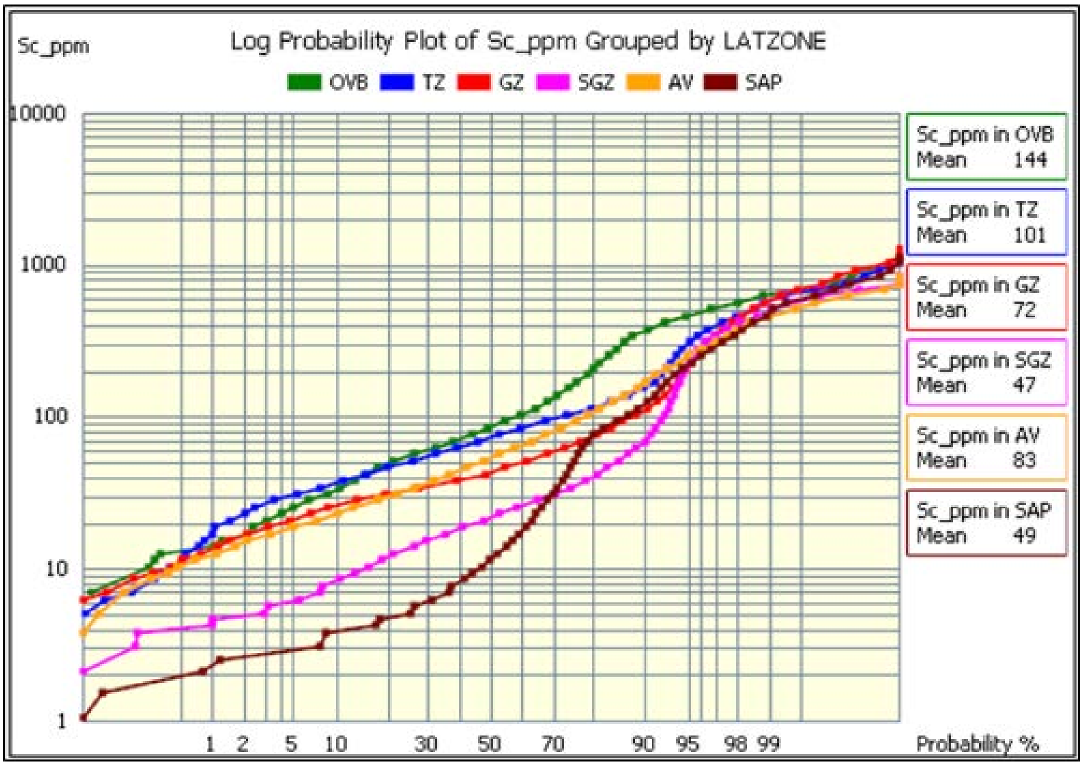


Figure 14-5: Scandium Distribution

However, there are also multiple distributions evident, as illustrated in the histograms in the overlays of log probability plots by LATZONE in Figure 14-6 to Figure 14-11..

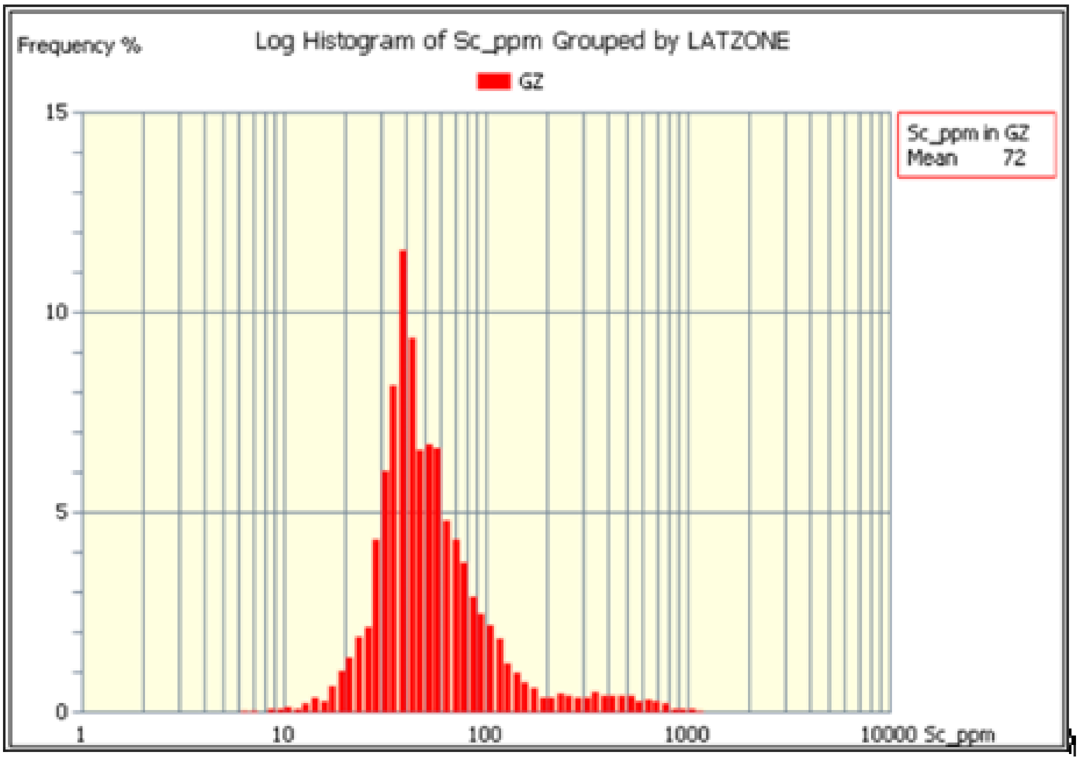


Figure 14-6: Scandium in GZ

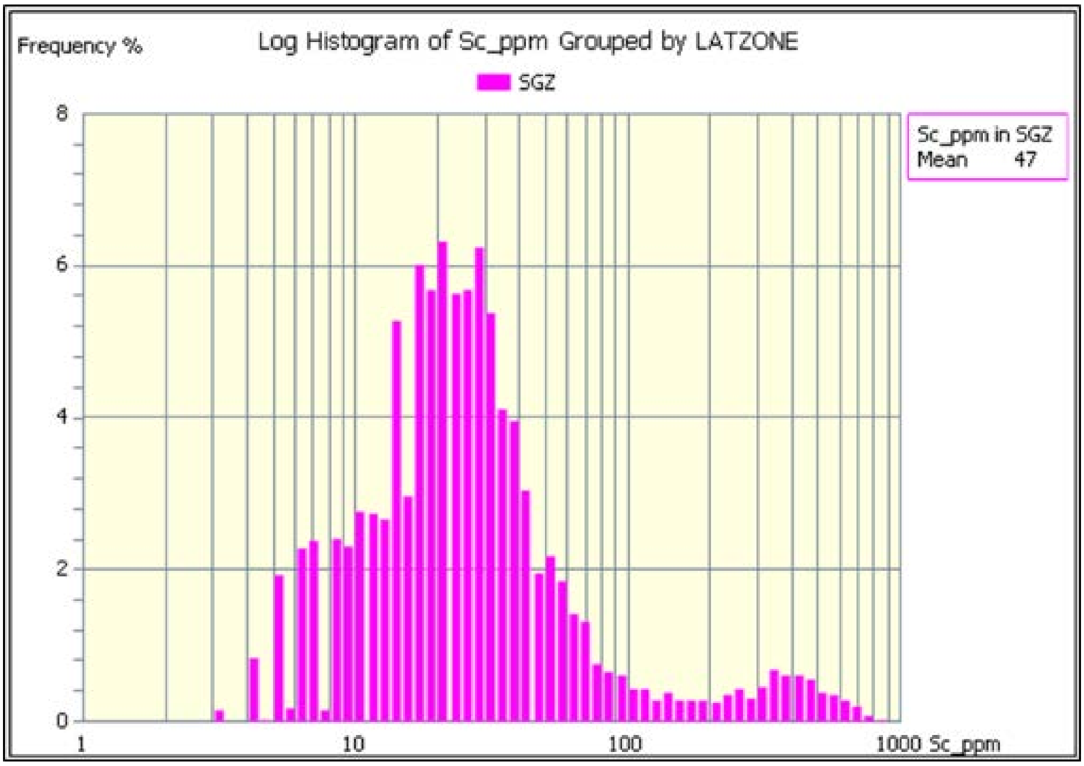


Figure 14-7: Scandium in SGZ

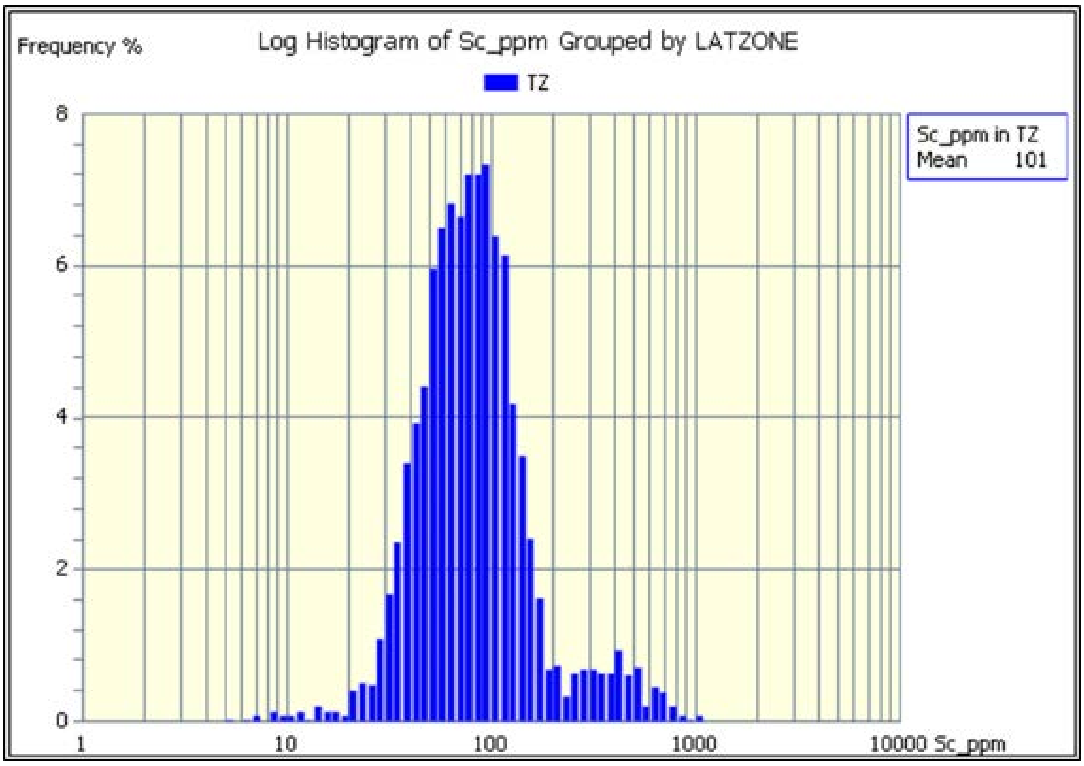


Figure 14-8: Scandium in TZ

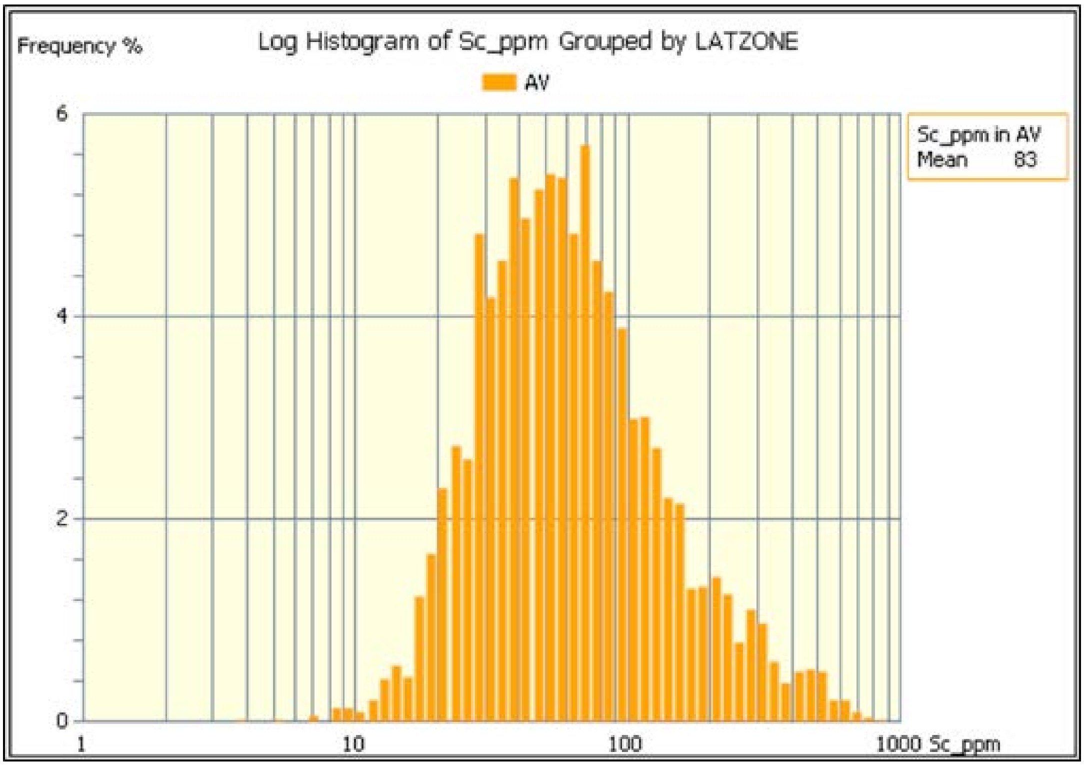


Figure 14-9: Scandium in AV

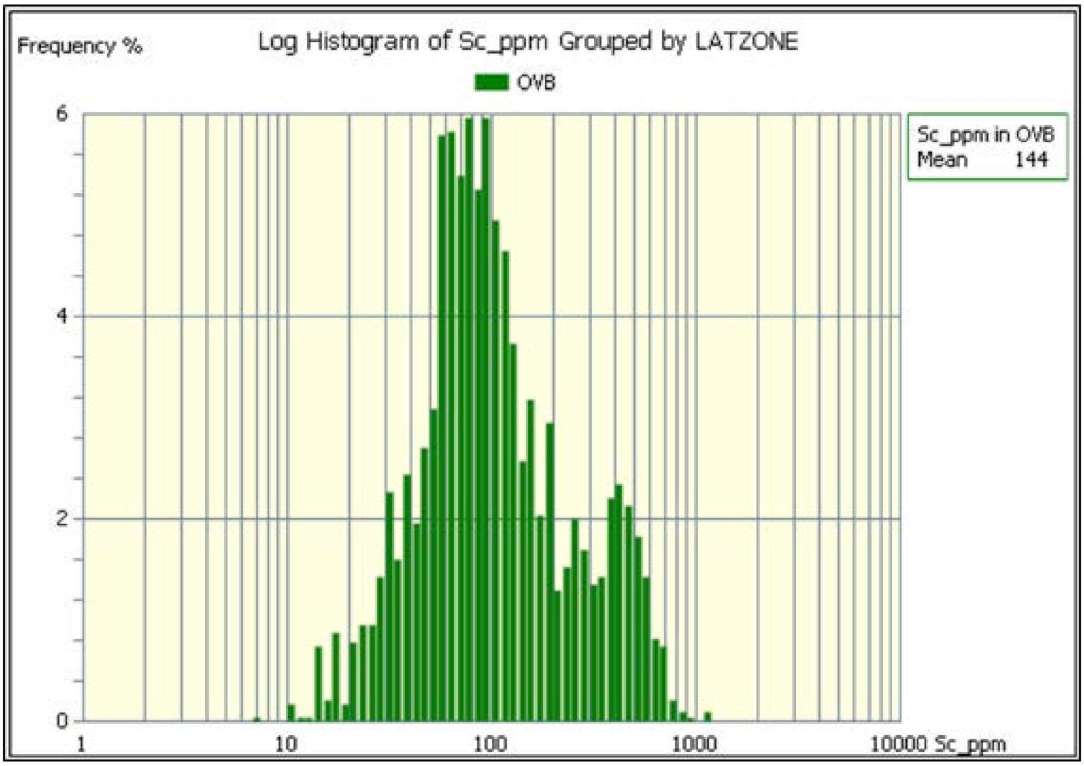


Figure 14-10: Scandium in OVB

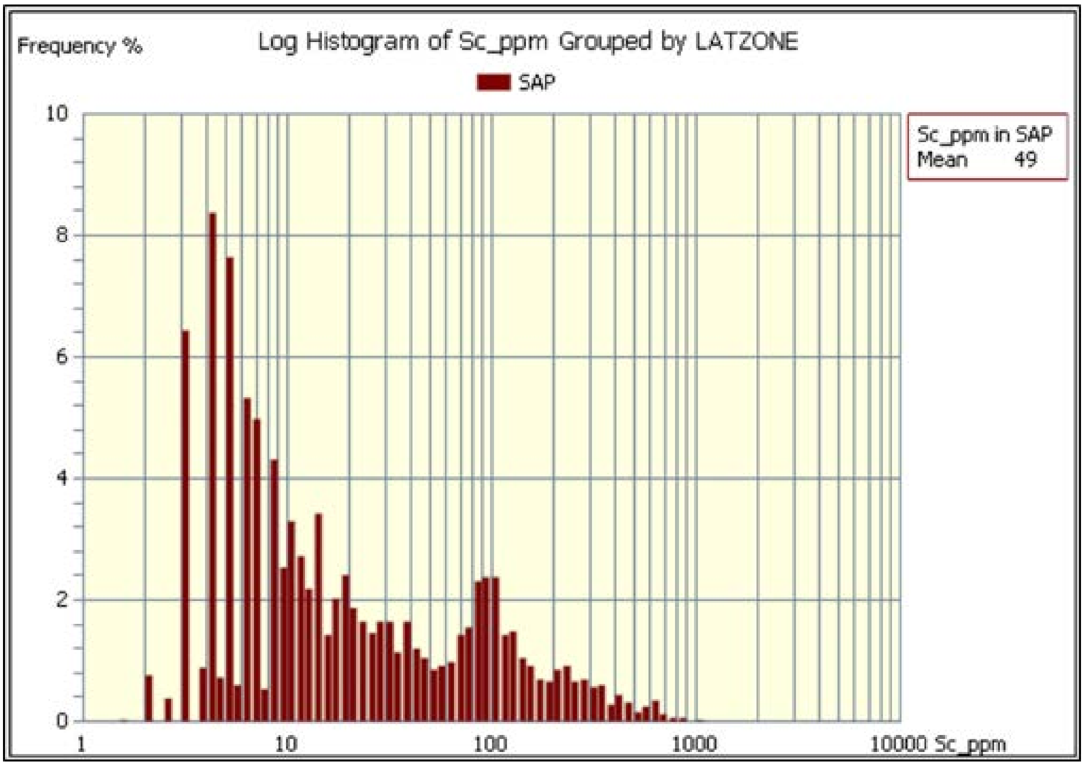


Figure 14-11: Scandium in SAP

As a result of the multiple population issue, an analysis was carried out on the spatial distribution of scandium, both globally (see Figure 14-12) and by LATZONE domains (see Figure 14-13 to Figure 14-17). 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 boundary of the serpentinised dunite complex (which is shown in white on following plans). 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 14.1.13 on scandium modelling.

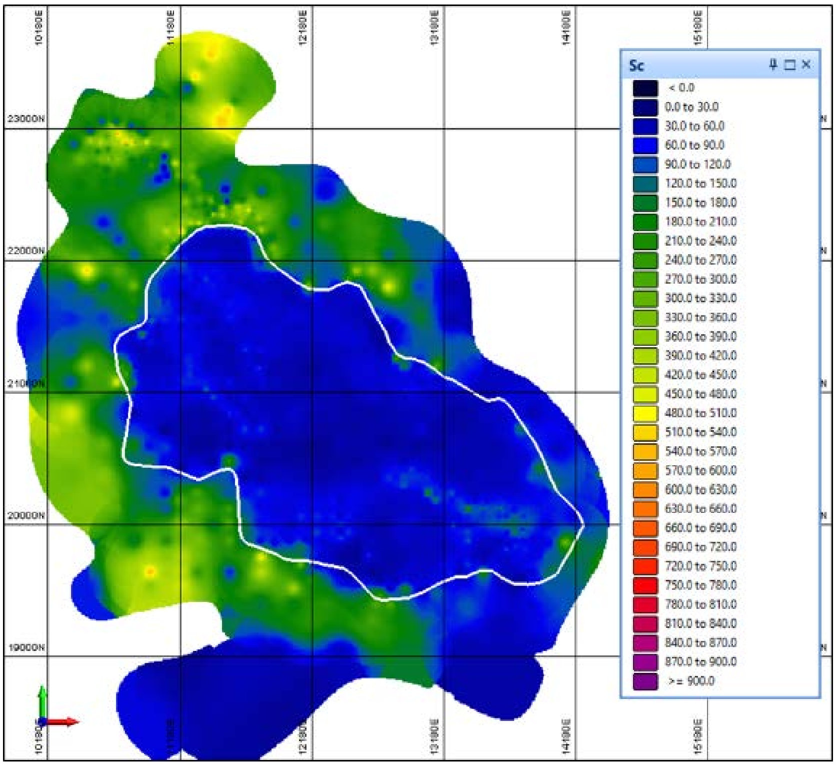


Figure 14-12: Global Scandium Distribution

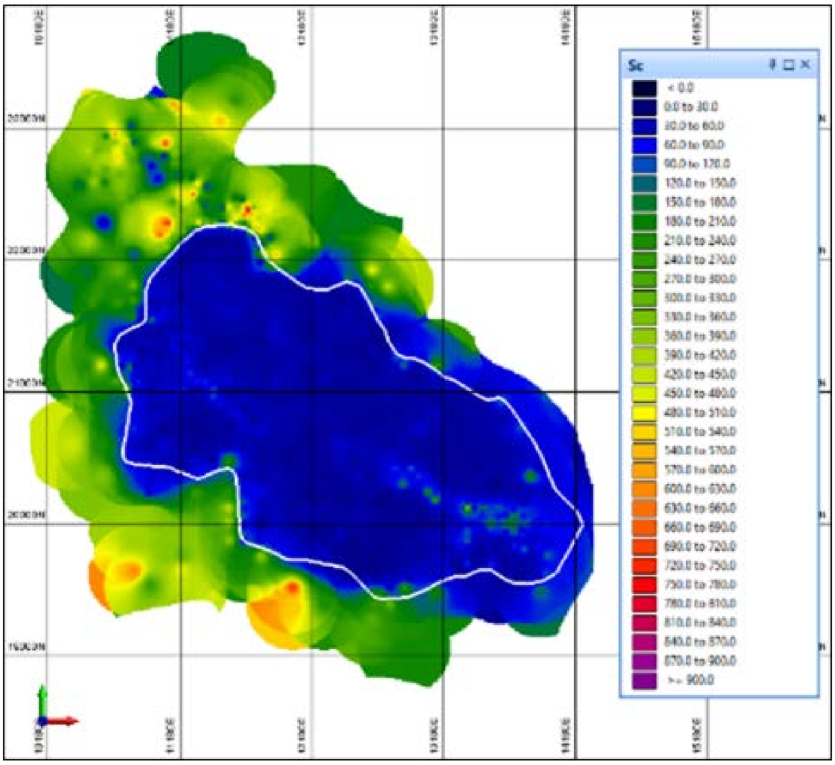


Figure 14-13: Scandium in GZ

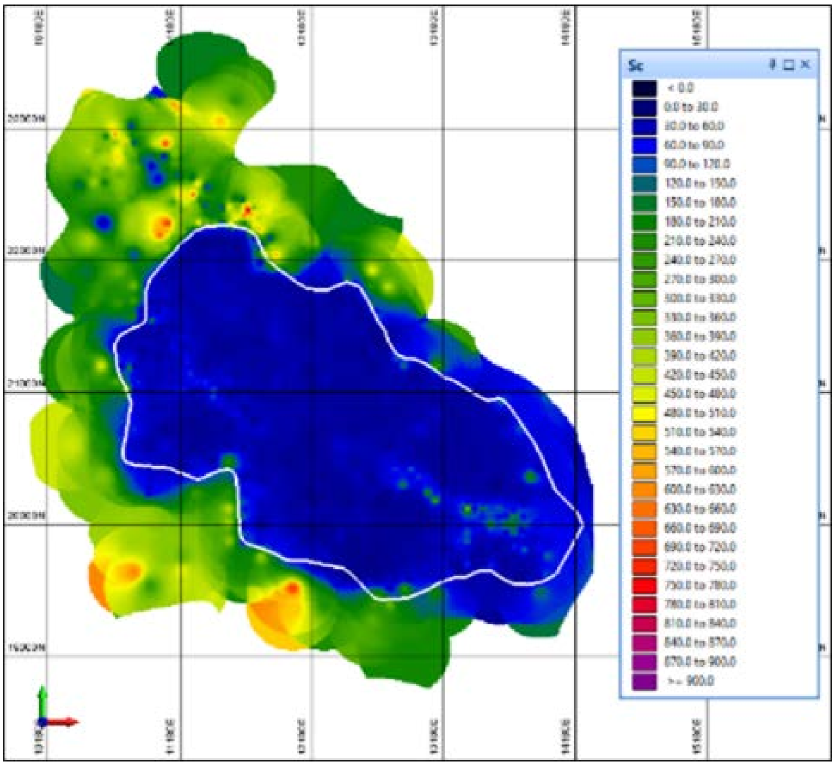


Figure 14-14: Scandium in SGZ

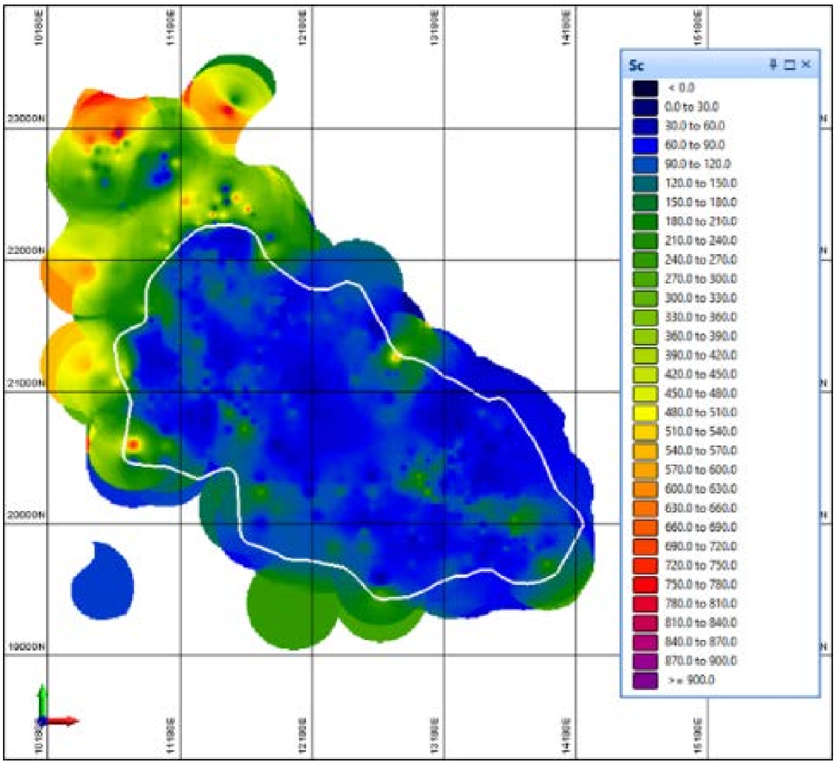


Figure 14-15: Scandium in TZ

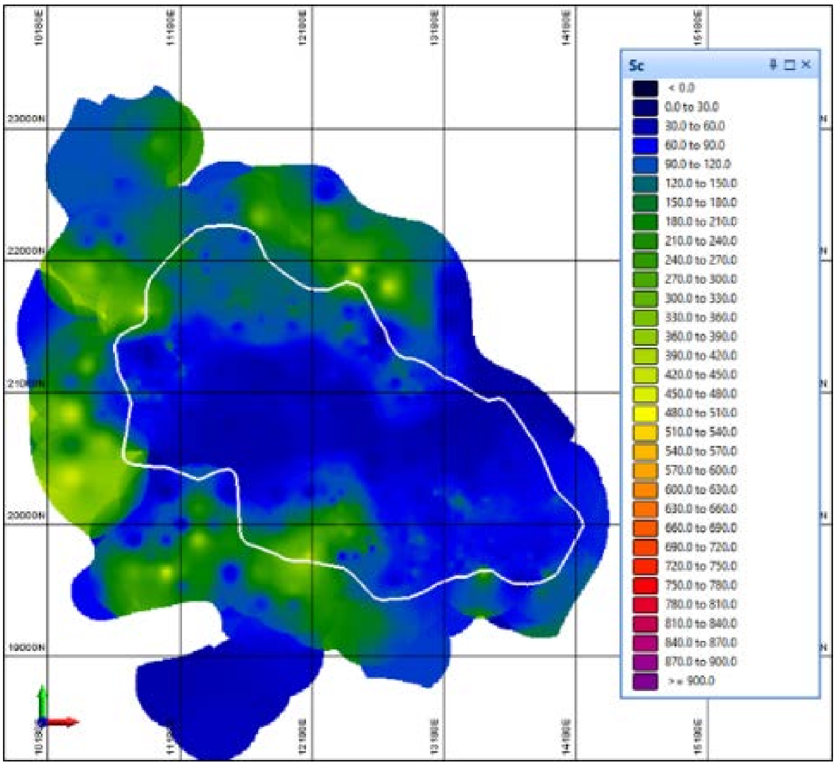


Figure 14-16: Scandium in AV

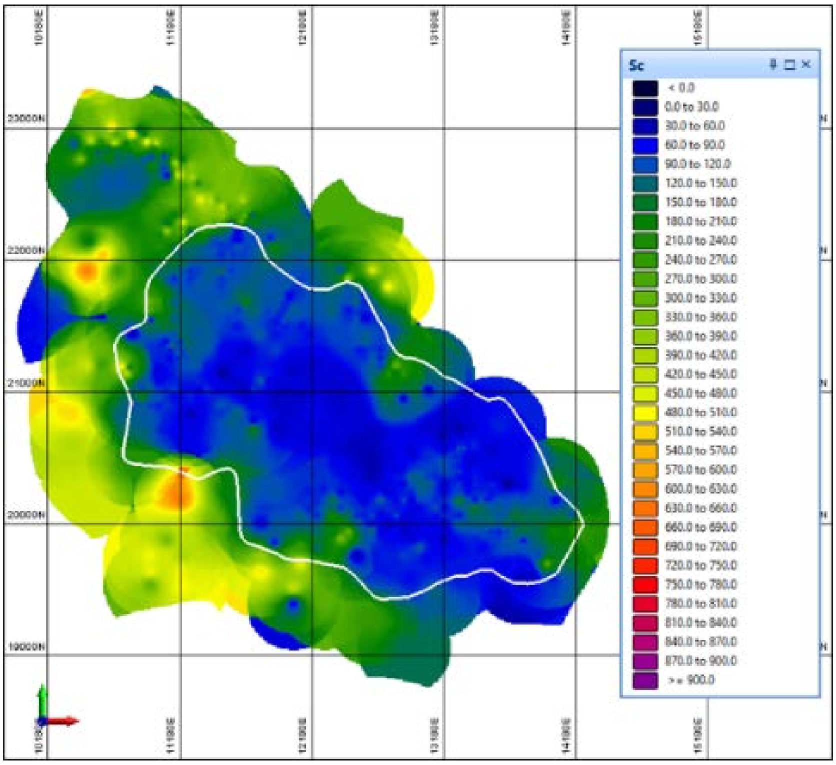


Figure 14-17: Scandium in OVB

When scandium distribution is separated into data above the dunite complex (see Figure 14-18), and data outside the dunite complex (see Figure 14-19), the differences are clear.

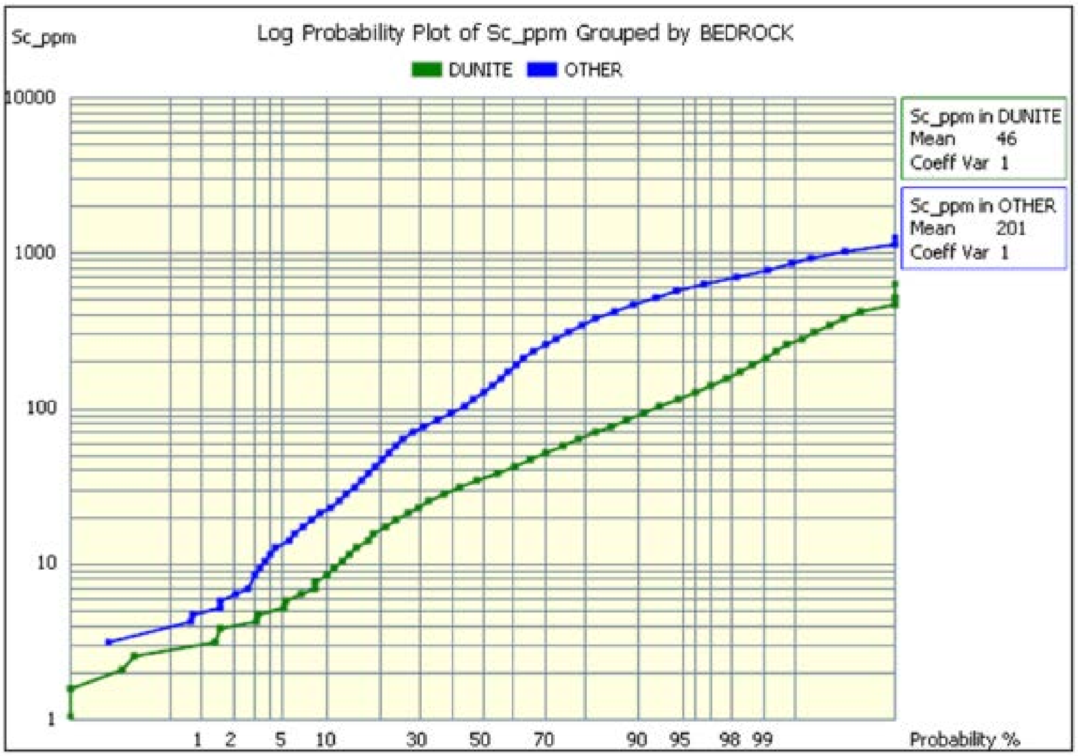


Figure 14-18: Scandium above Dunite Complex

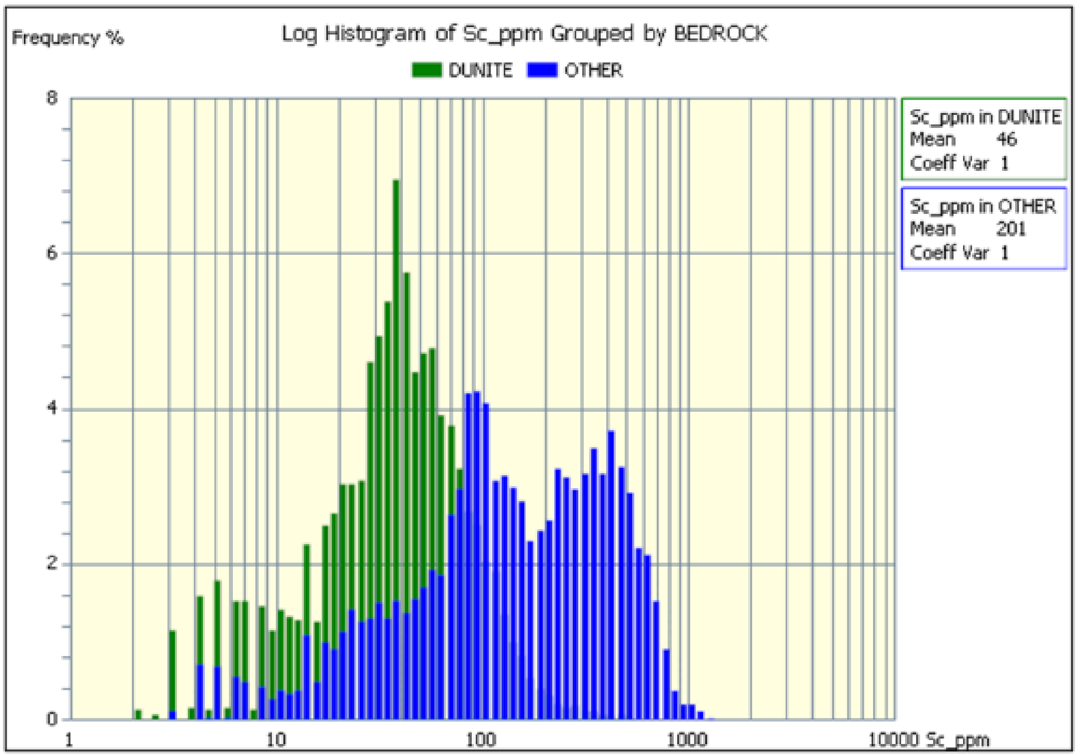


Figure 14-19: Data outside Dunite Complex

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.

14.1.4 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 Table 14-4 and Table 14-5. All others were uncut.

Table 14-4: Applied Top-Cuts

High-Grade Caps by LATZONE		
LATZONE	**Nickel (%)**	**Cobalt (%)**
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

Table 14-5: Top-Cut Minor Variables

Element	Minor Variable Concentration (ppm)
Scandium	900
Platinum	30
Lead	600
Gold	600
Manganese	15,000
Zinc	1,500
Copper	6,000
Chromium	10,000
Arsenic	70

The overlay log probability plots shown in Figure 14-20 to Figure 14-32 below show the basis of the caps and again illustrate that the LATZONE domaining for almost all elements shows clearly different distributions.

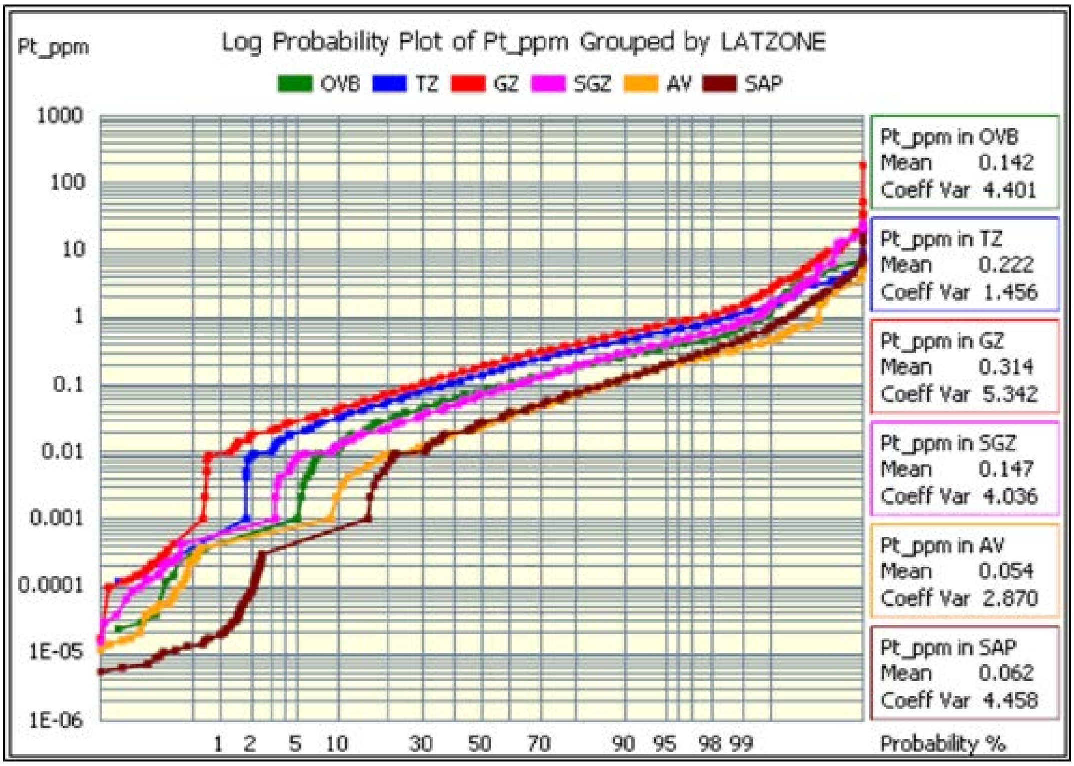


Figure 14-20: Log probability plot of Platinum

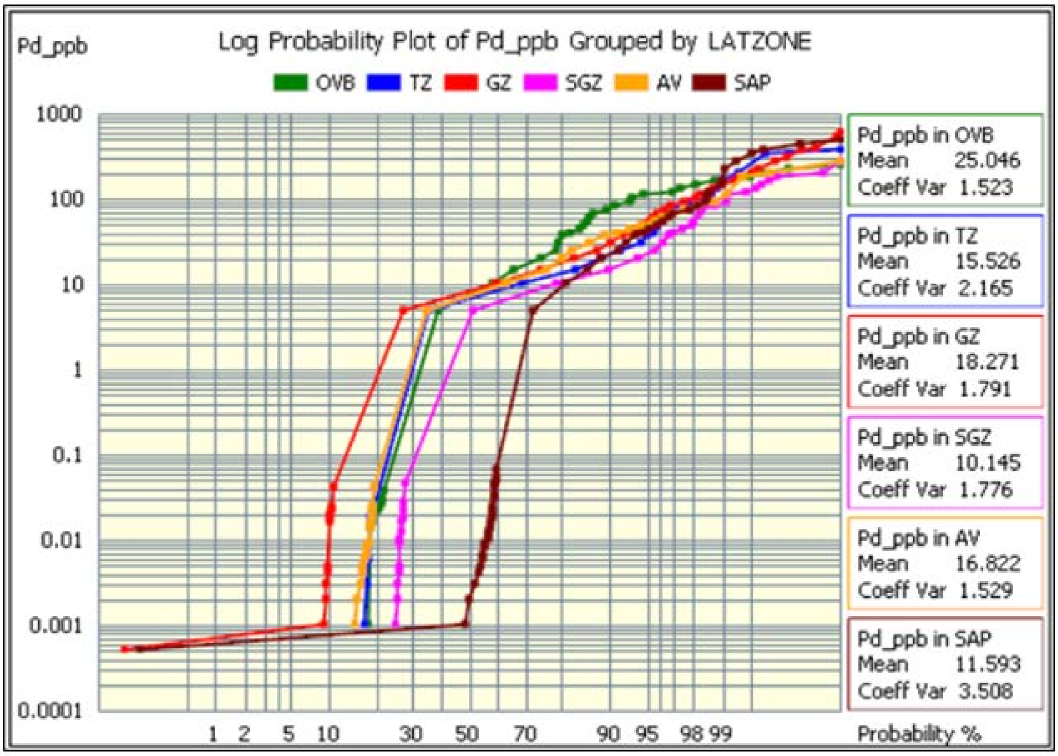


Figure 14-21: Log probability plot of Lead

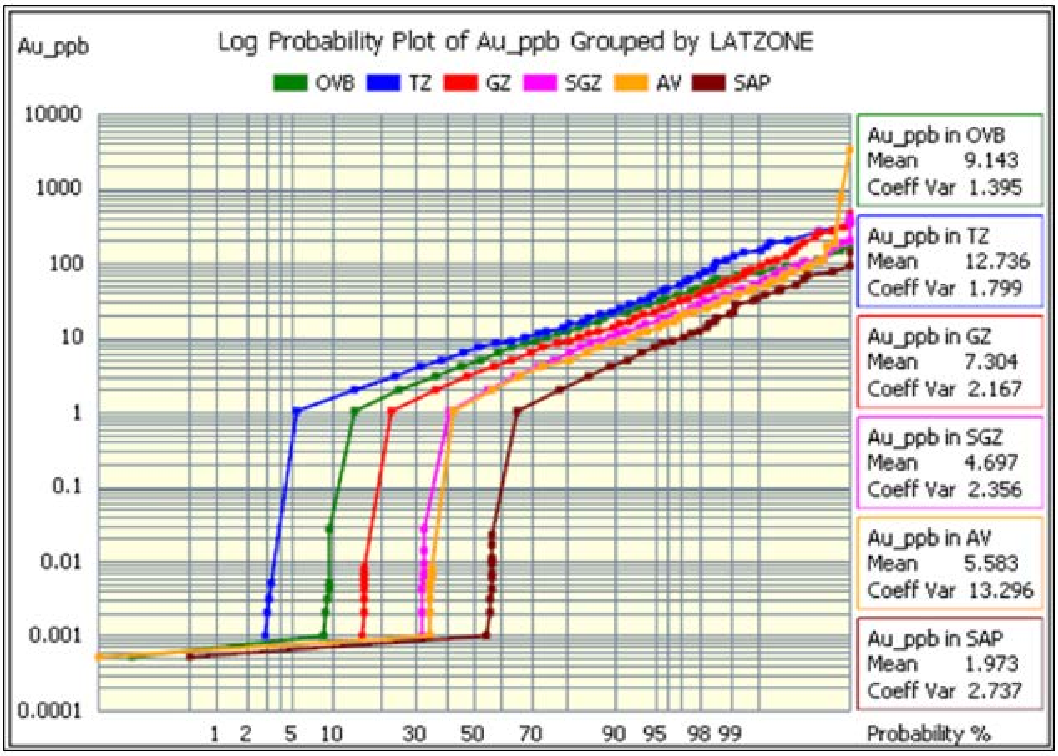


Figure 14-22: Log probability plot of Gold

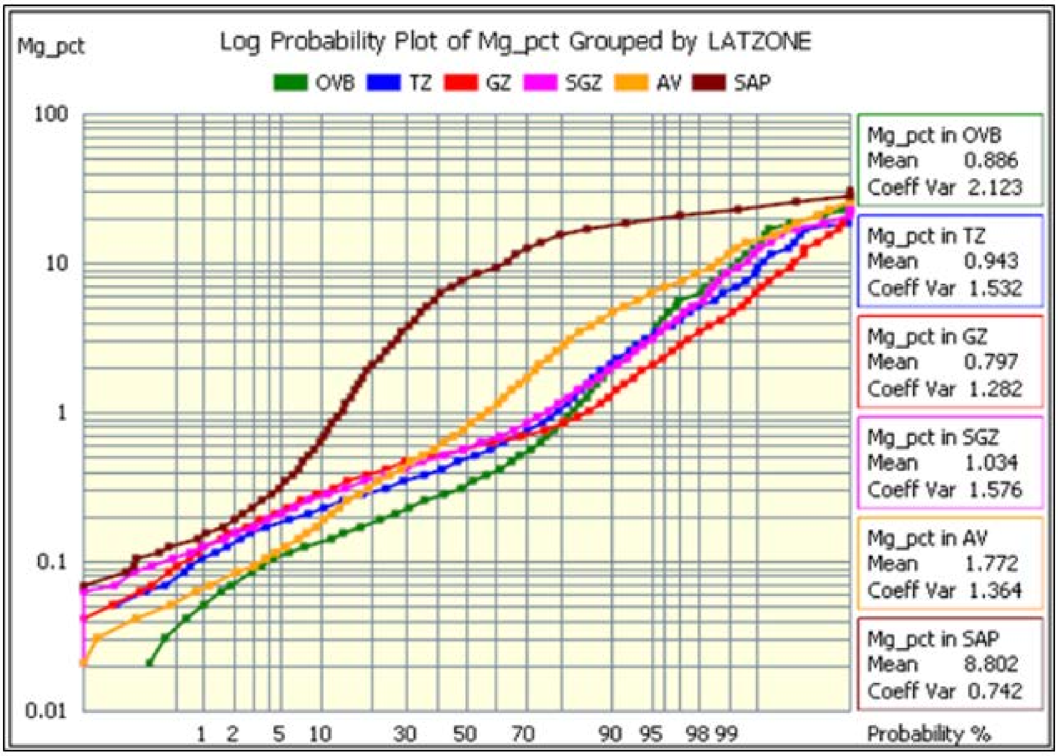


Figure 14-23: Log probability plot of Magnesium

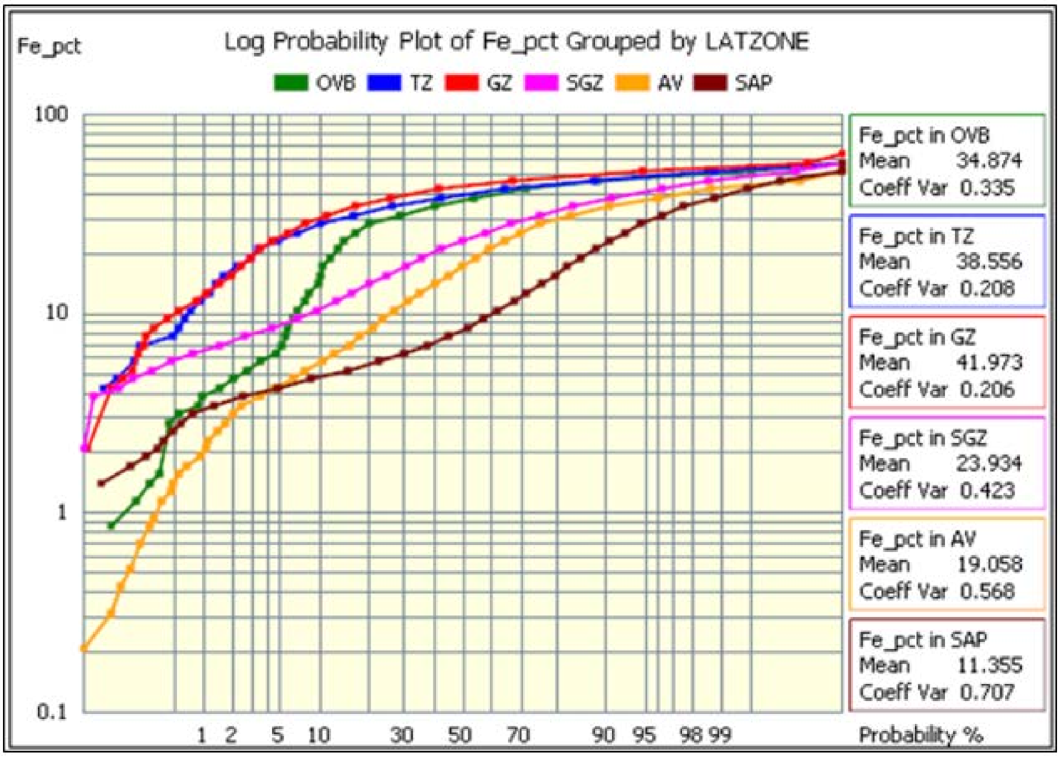


Figure 14-24: Log probability plot of Iron

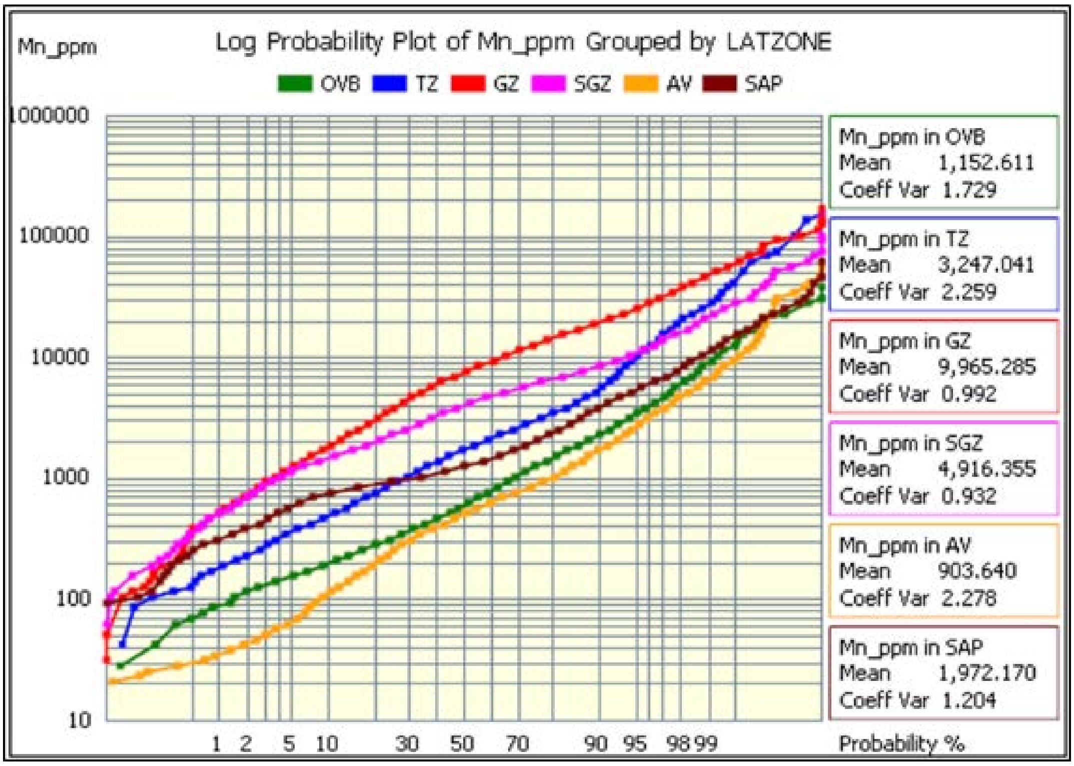


Figure 14-25: Log probability plot of Manganese

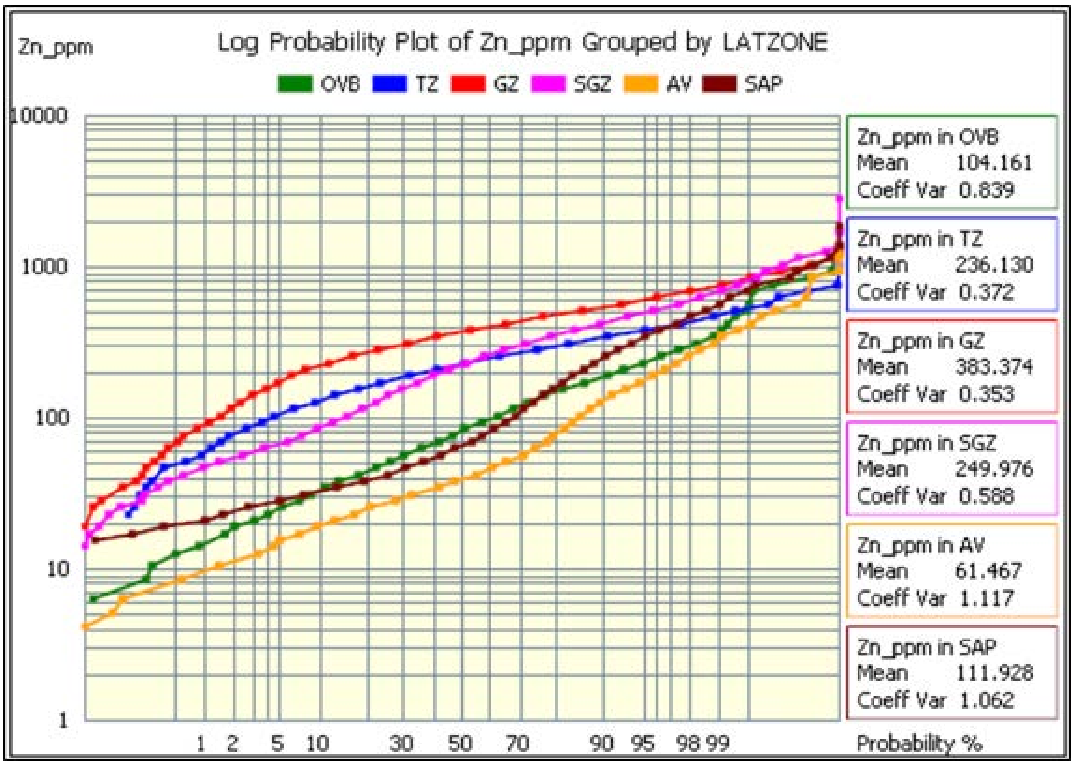


Figure 14-26: Log probability plot of Zinc

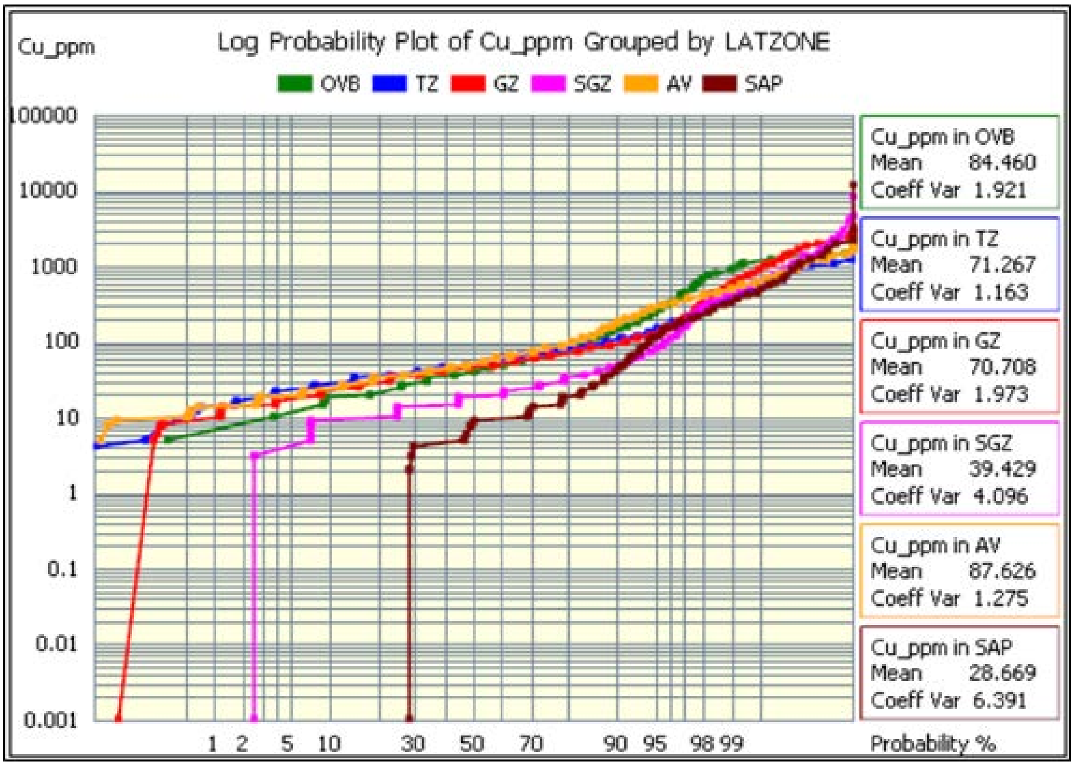


Figure 14-27: Log probability plot of Copper

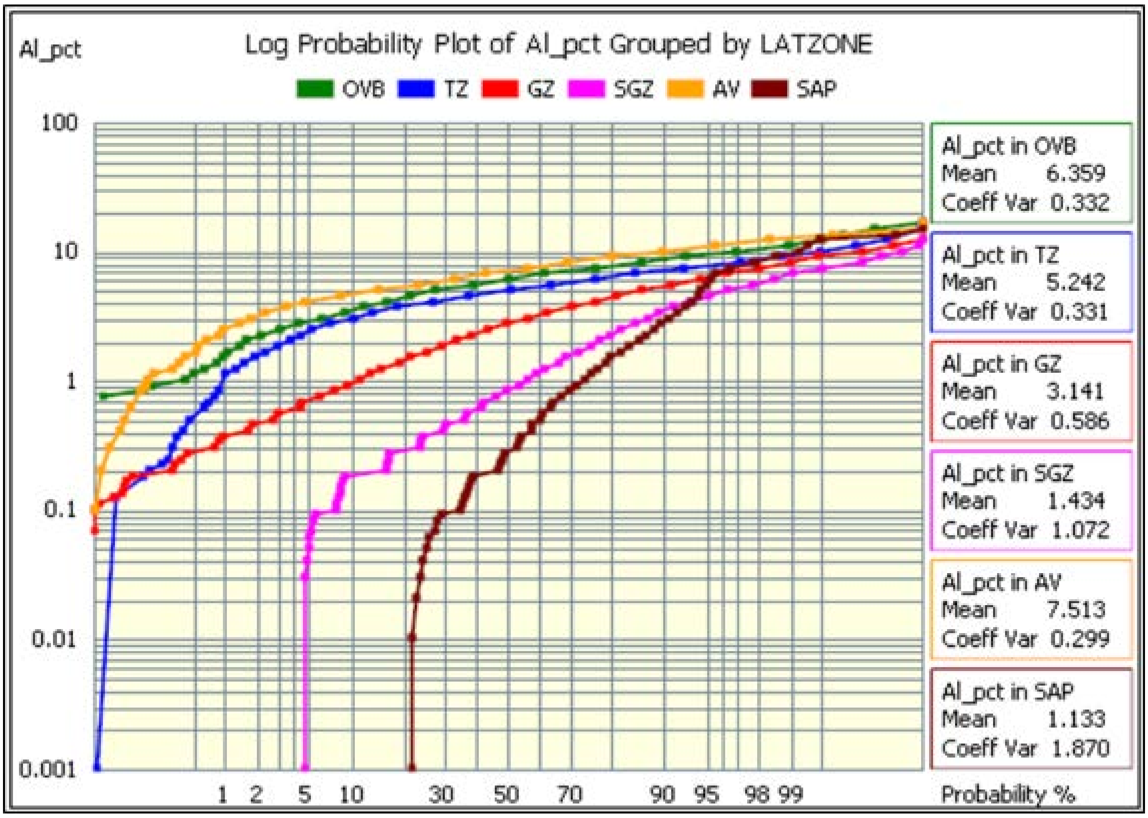


Figure 14-28: Log probability plot of Aluminium

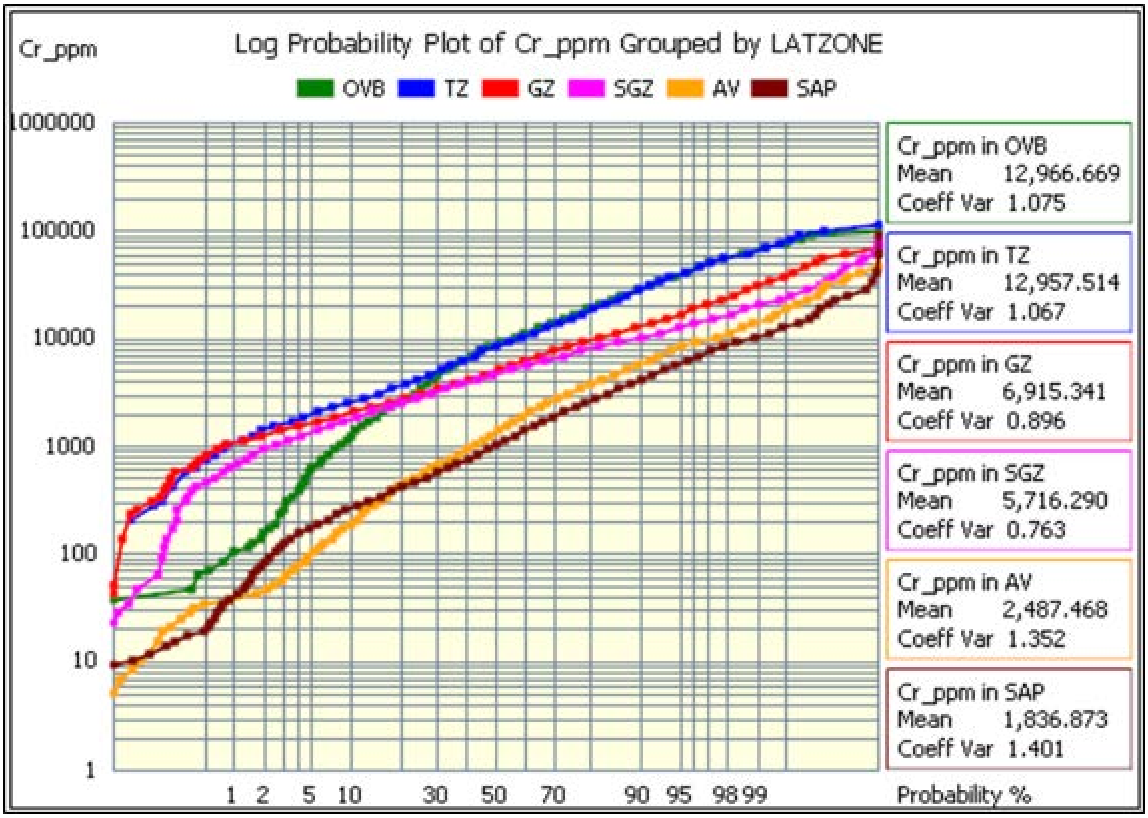


Figure 14-29: Log probability plot of Chromium

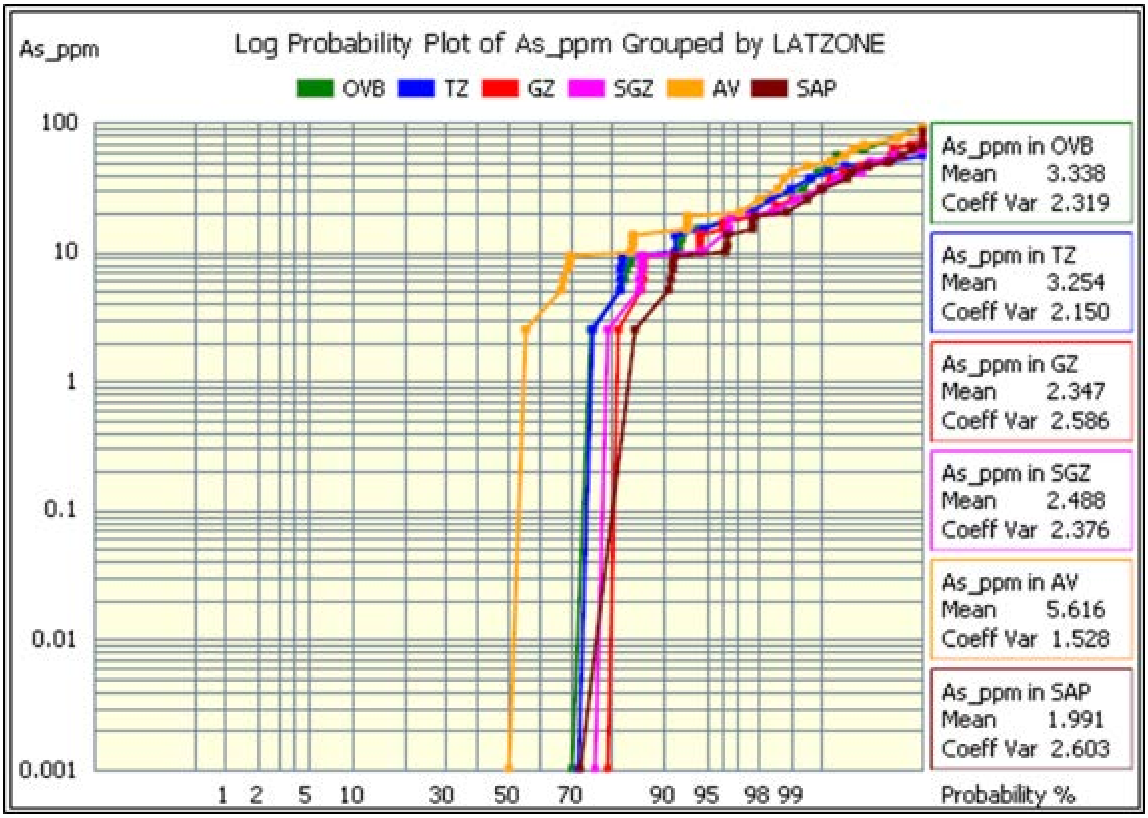


Figure 14-30: Log probability plot of Arsenic

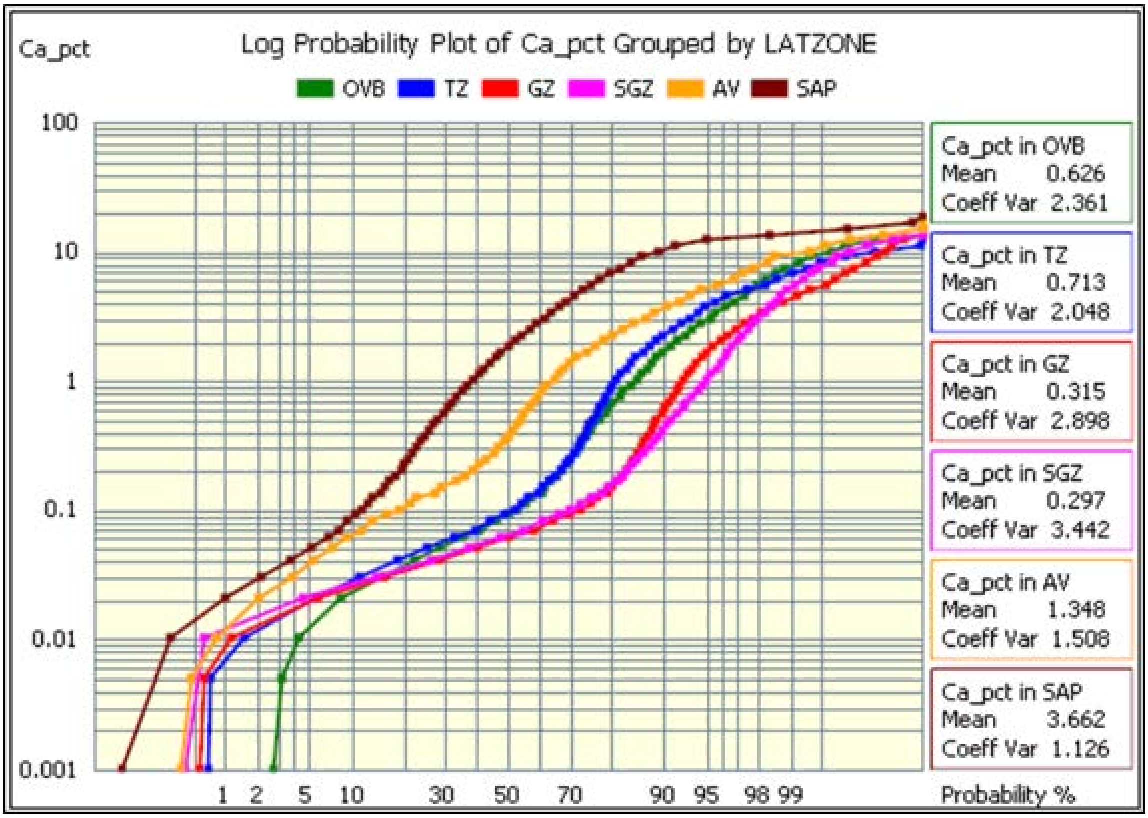


Figure 14-31: Log probability plot of Calcium

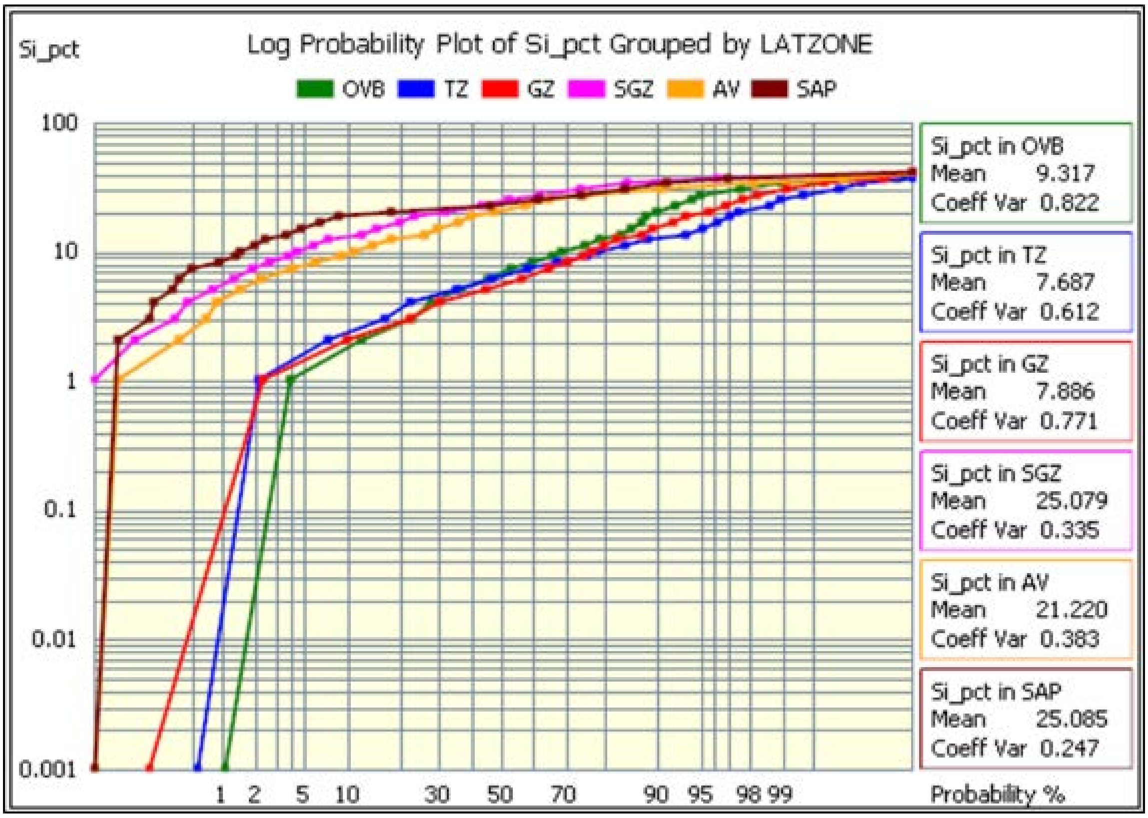


Figure 14-32: Log probability plot of Silica

14.1.5 Domain Boundary Analysis

While it is clear from overlay plots of the distribution of all elements by LATZONE (see an example), a boundary analysis (see Figure 14-33 to Figure 14-40) 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 nickel, cobalt, scandium, platinum, iron, silicon, aluminium and manganese). It has been concluded that hard wiring of boundaries is appropriate.

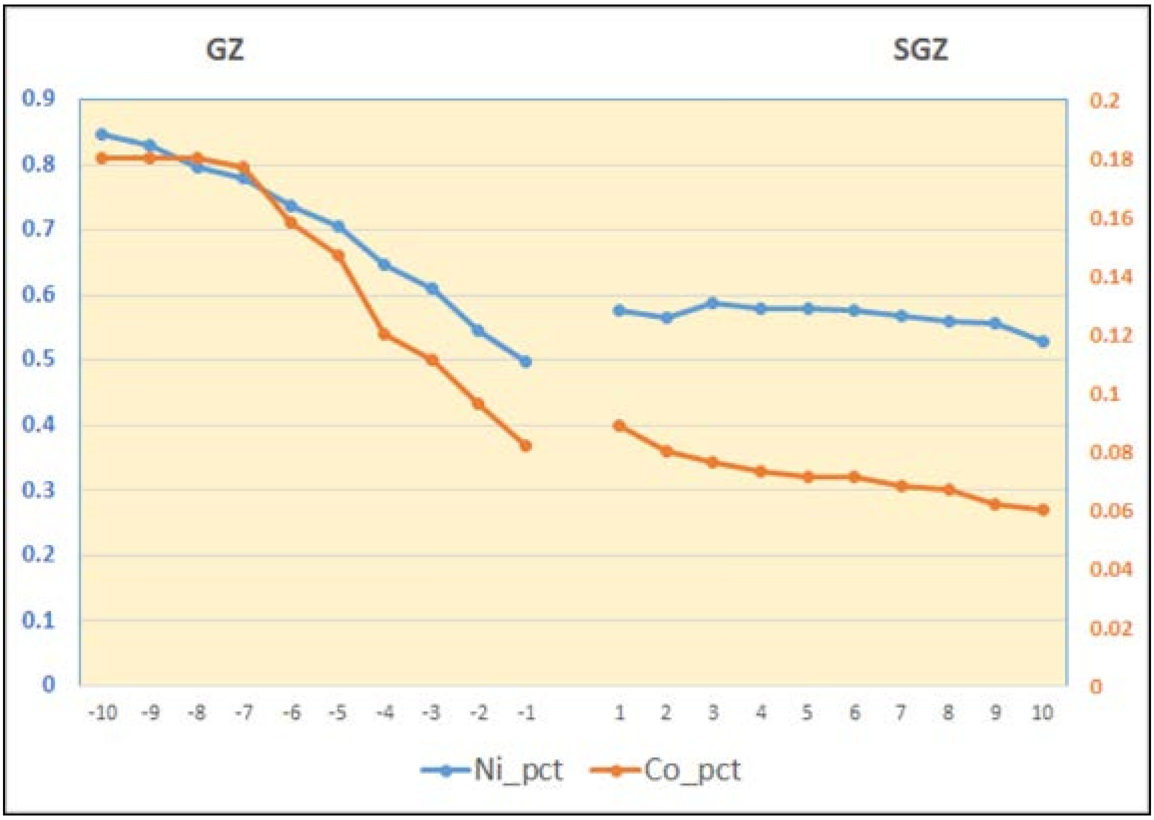


Figure 14-33: Boundary Analysis GZ/SGZ Ni and Co

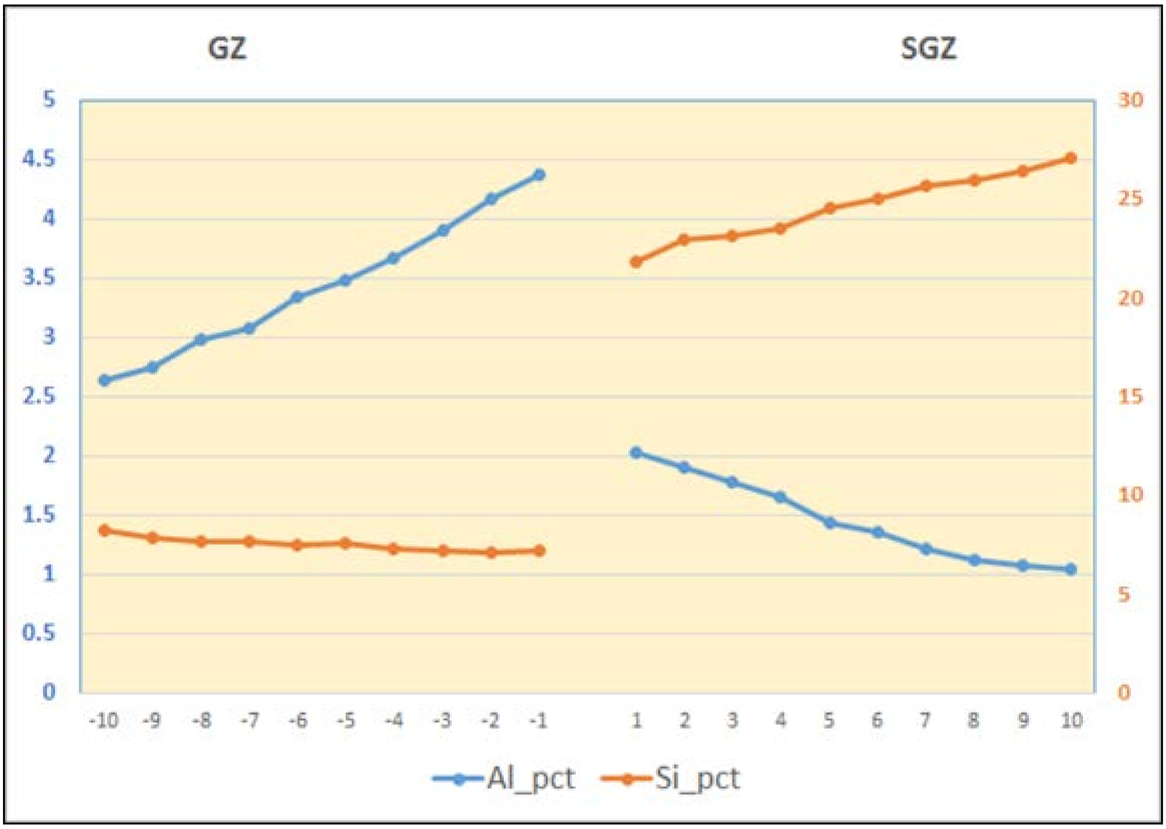


Figure 14-34: Boundary Analysis GZ/SGZ Al and Si

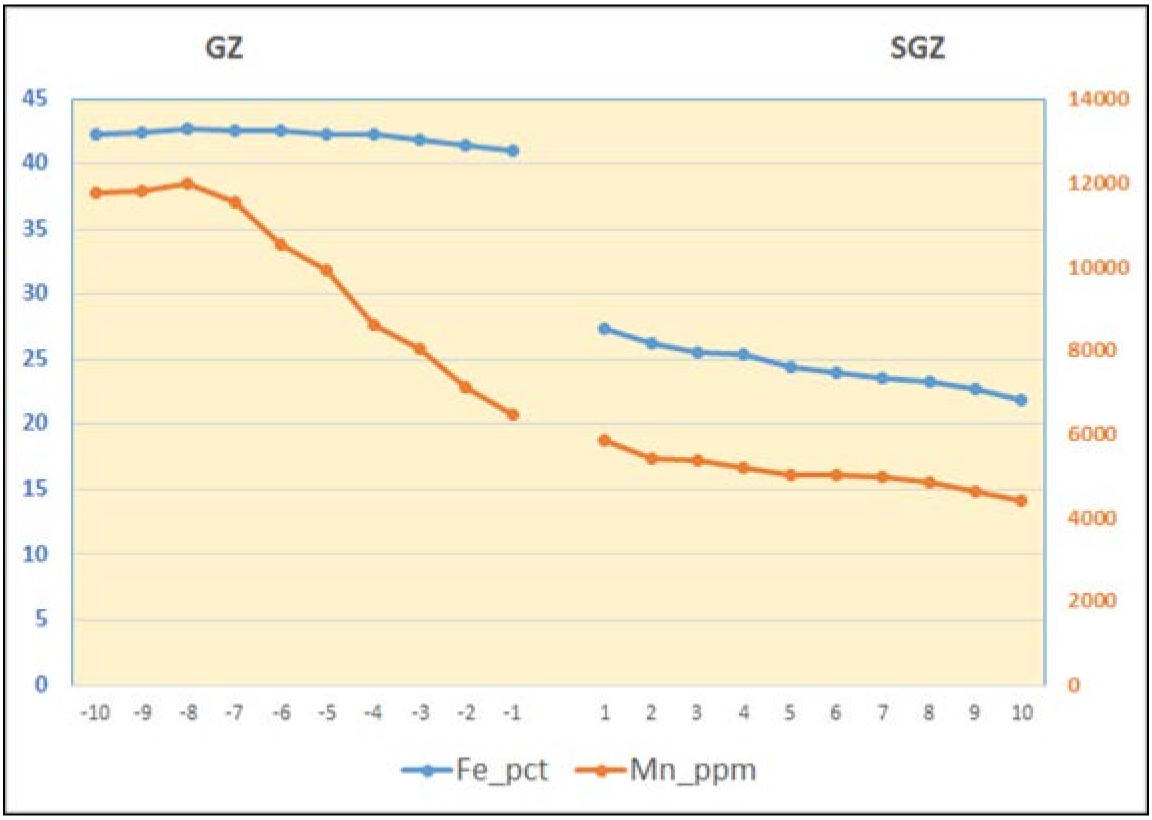


Figure 14-35: Boundary Analysis GZ/SGZ Fe and Mn

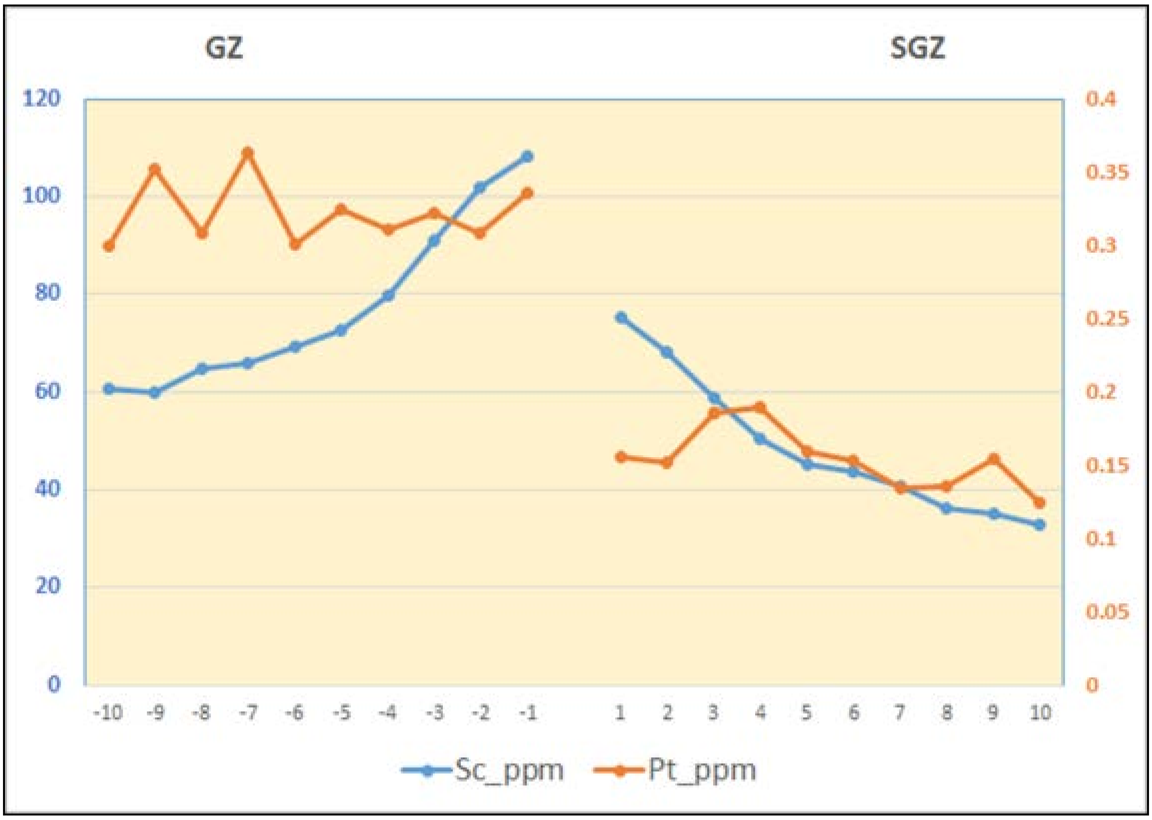


Figure 14-36: Boundary Analysis GZ/SGZ Sc and Pt

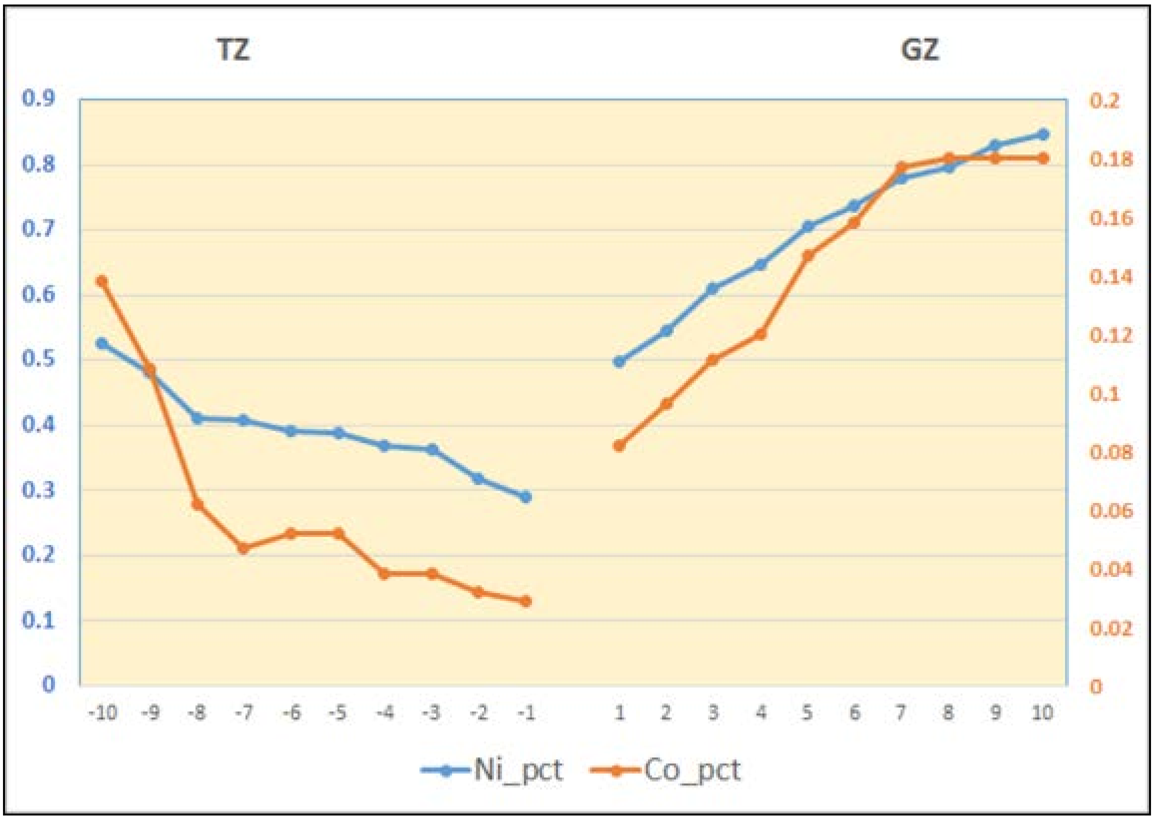


Figure 14-37: Boundary Analysis TZ/GZ Ni and Co

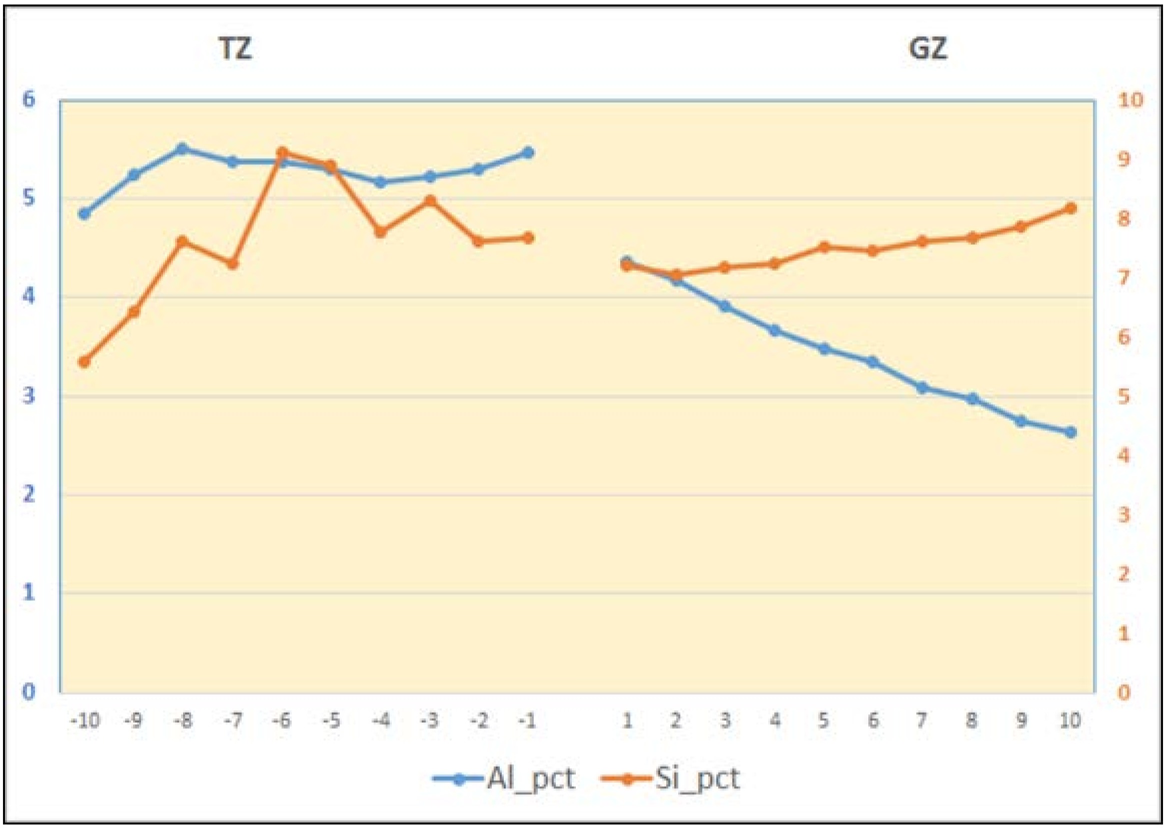


Figure 14-38: Boundary Analysis TZ/GZ Al and Si

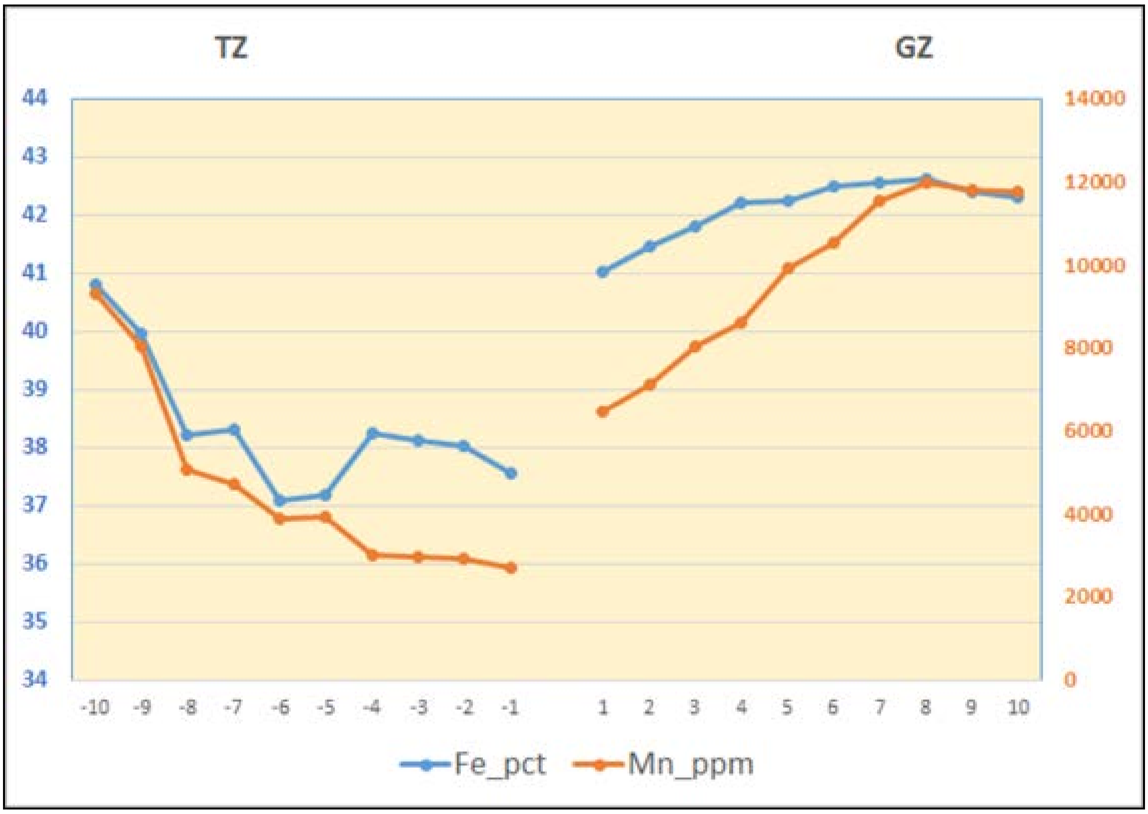


Figure 14-39: Boundary Analysis TZ/GZ Fe and Mn

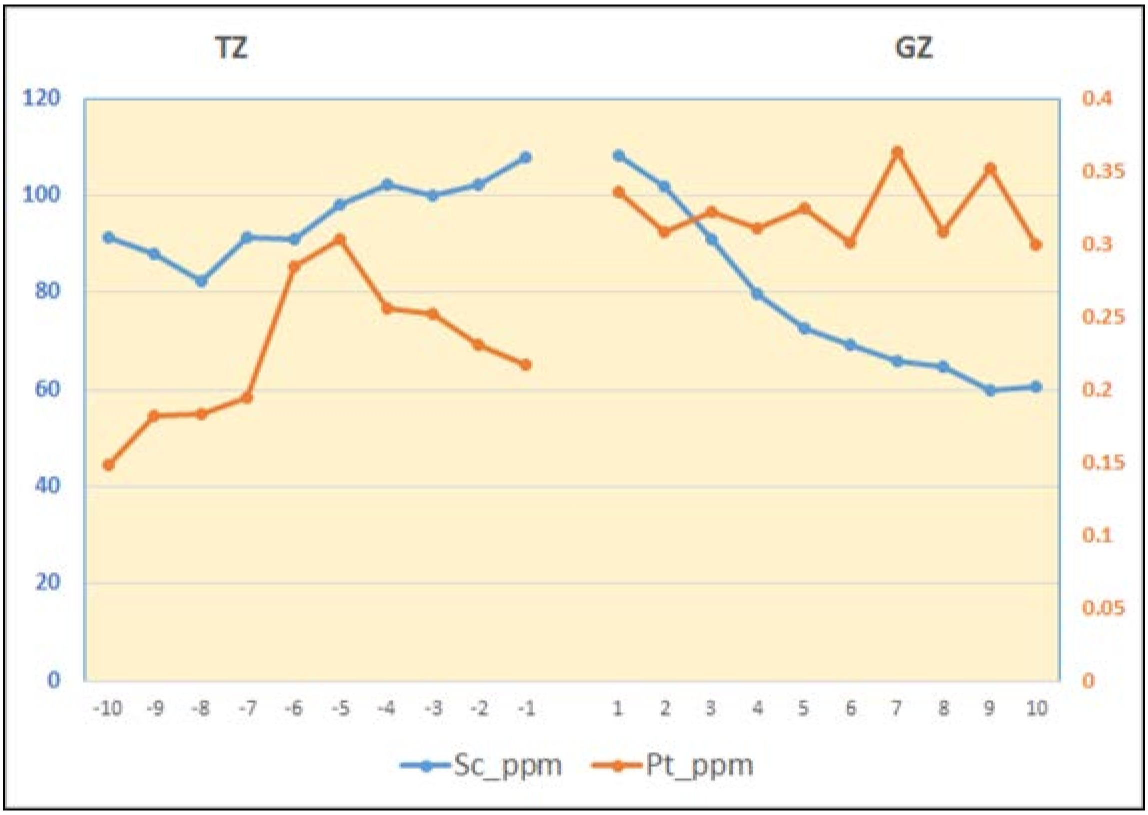
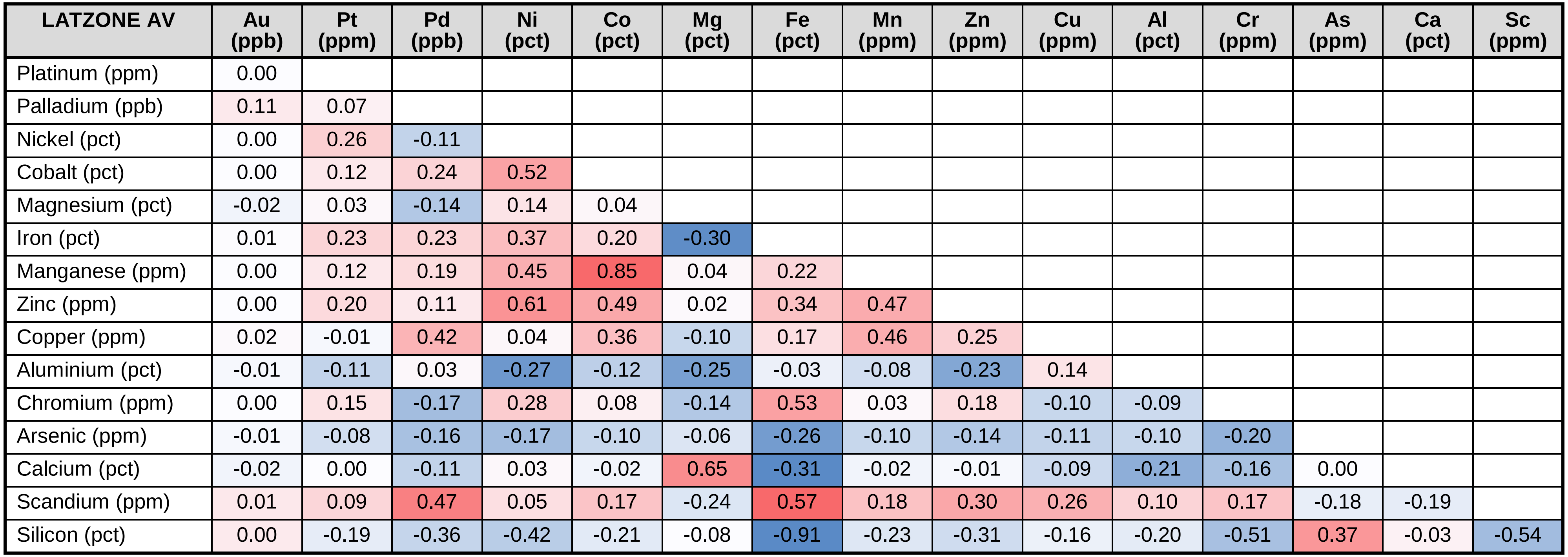


Figure 14-40: Boundary Analysis TZ/GZ Sc and Pt

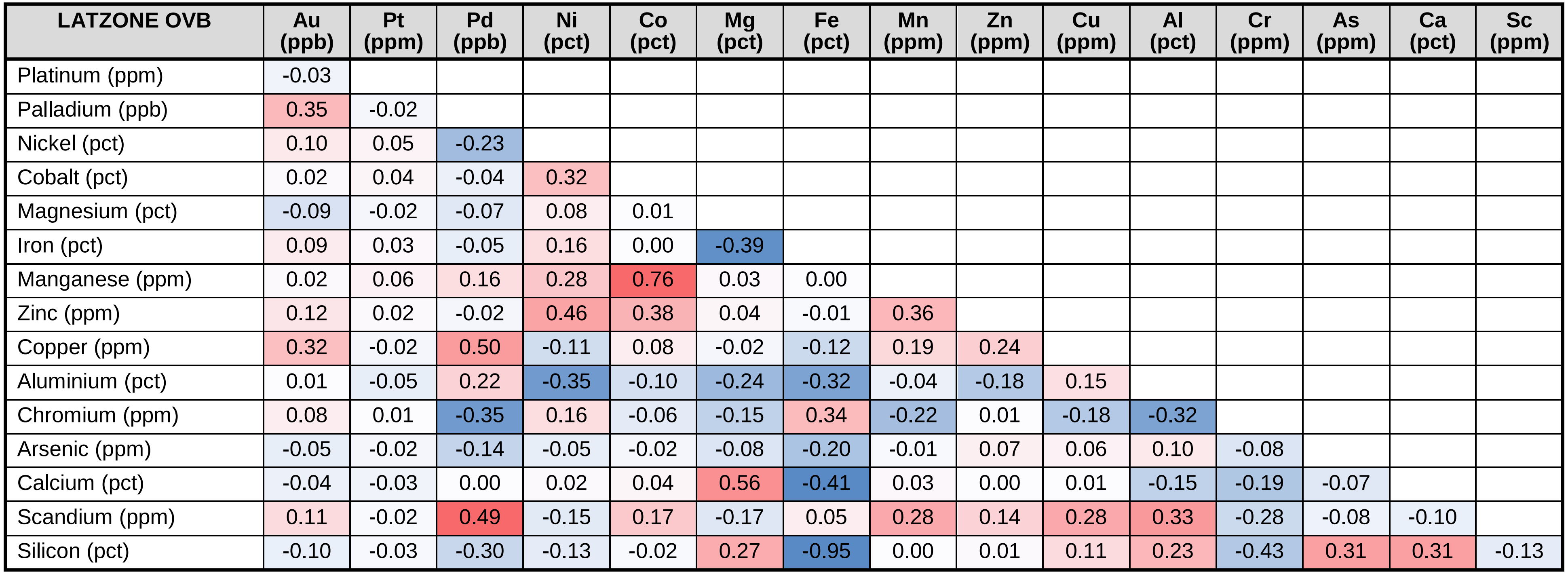
14.1.6 Correlation Analysis

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 platinum/palladium, iron/silicon, calcium/magnesium, cobalt/manganese. Correlation matrices are illustrated in Figure 14-41 to Figure 14-45.



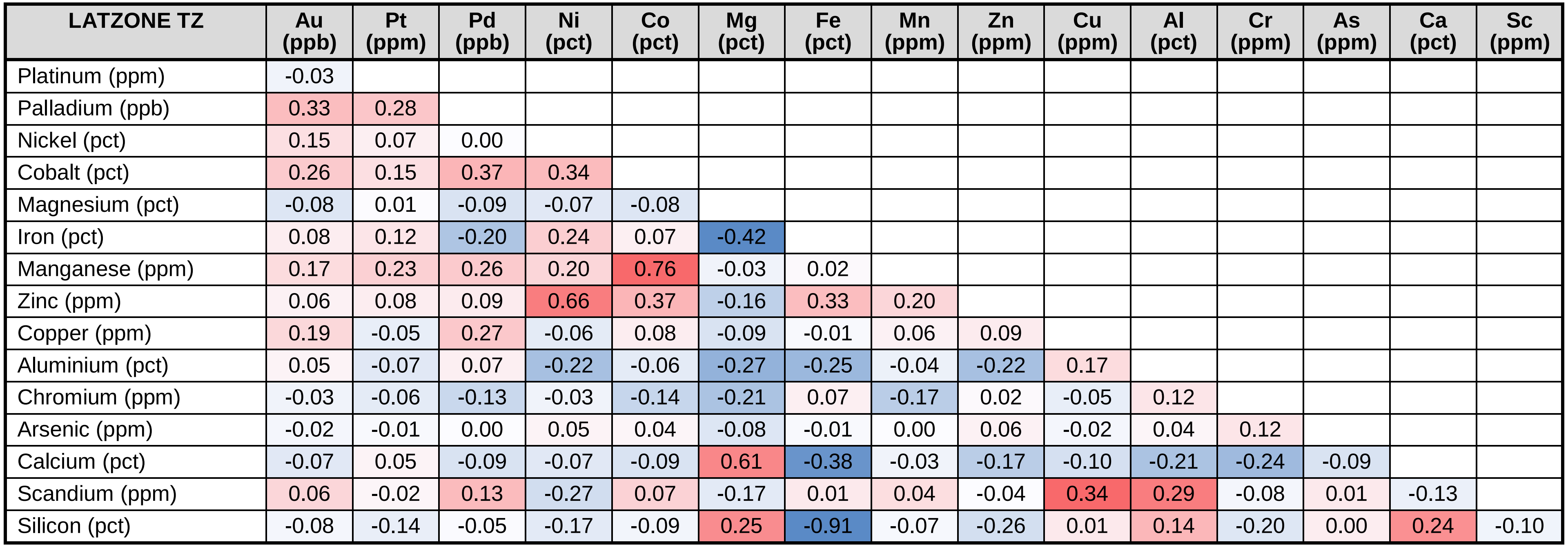
LATZONE AV	Au (ppb)	Pt (ppm)	Pd (ppb)	Ni (pct)	Co (pct)	Mg (pct)	Fe (pct)	Mn (ppm)	Zn (ppm)	Cu (ppm)	Al (pct)	Cr (ppm)	As (ppm)	Ca (pct)	Sc (ppm)
Platinum (ppm)	0.00														
Palladium (ppb)	0.11	0.07													
Nickel (pct)	0.00	0.26	-0.11												
Cobalt (pct)	0.00	0.12	0.24	0.52											
Magnesium (pct)	-0.02	0.03	-0.14	0.14	0.04										
Iron (pct)	0.01	0.23	0.23	0.37	0.20	-0.30									
Manganese (ppm)	0.00	0.12	0.19	0.45	0.85	0.04	0.22								
Zinc (ppm)	0.00	0.20	0.11	0.61	0.49	0.02	0.34	0.47							
Copper (ppm)	0.02	-0.01	0.42	0.04	0.36	-0.10	0.17	0.46	0.25						
Aluminium (pct)	-0.01	-0.11	0.03	-0.27	-0.12	-0.25	-0.03	-0.08	-0.23	0.14					
Chromium (ppm)	0.00	0.15	-0.17	0.28	0.08	-0.14	0.53	0.03	0.18	-0.10	-0.09				
Arsenic (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			
Calcium (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		
Scandium (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	
Silicon (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 14-41: 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)
Platinum (ppm)	-0.03														
Palladium (ppb)	0.35	-0.02													
Nickel (pct)	0.10	0.05	-0.23												
Cobalt (pct)	0.02	0.04	-0.04	0.32											
Magnesium (pct)	-0.09	-0.02	-0.07	0.08	0.01										
Iron (pct)	0.09	0.03	-0.05	0.16	0.00	-0.39									
Manganese (ppm)	0.02	0.06	0.16	0.28	0.76	0.03	0.00								
Zinc (ppm)	0.12	0.02	-0.02	0.46	0.38	0.04	-0.01	0.36							
Copper (ppm)	0.32	-0.02	0.50	-0.11	0.08	-0.02	-0.12	0.19	0.24						
Aluminium (pct)	0.01	-0.05	0.22	-0.35	-0.10	-0.24	-0.32	-0.04	-0.18	0.15					
Chromium (ppm)	0.08	0.01	-0.35	0.16	-0.06	-0.15	0.34	-0.22	0.01	-0.18	-0.32				
Arsenic (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			
Calcium (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		
Scandium (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	
Silicon (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 14-42: Correlation Matrix OVB

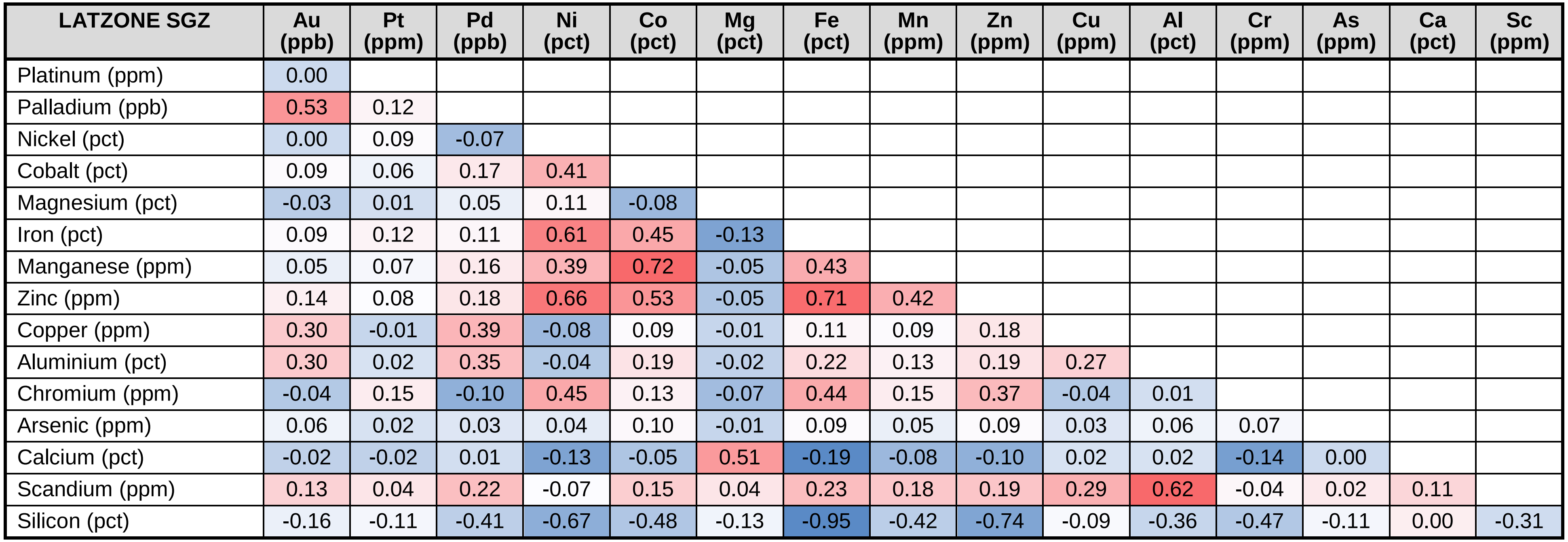


LATZONE TZ	Au (ppb)	Pt (ppm)	Pd (ppb)	Ni (pct)	Co (pct)	Mg (pct)	Fe (pct)	Mn (ppm)	Zn (ppm)	Cu (ppm)	Al (pct)	Cr (ppm)	As (ppm)	Ca (pct)	Sc (ppm)
Platinum (ppm)	-0.03														
Palladium (ppb)	0.33	0.28													
Nickel (pct)	0.15	0.07	0.00												
Cobalt (pct)	0.26	0.15	0.37	0.34											
Magnesium (pct)	-0.08	0.01	-0.09	-0.07	-0.08										
Iron (pct)	0.08	0.12	-0.20	0.24	0.07	-0.42									
Manganese (ppm)	0.17	0.23	0.26	0.20	0.76	-0.03	0.02								
Zinc (ppm)	0.06	0.08	0.09	0.66	0.37	-0.16	0.33	0.20							
Copper (ppm)	0.19	-0.05	0.27	-0.06	0.08	-0.09	-0.01	0.06	0.09						
Aluminium (pct)	0.05	-0.07	0.07	-0.22	-0.06	-0.27	-0.25	-0.04	-0.22	0.17					
Chromium (ppm)	-0.03	-0.06	-0.13	-0.03	-0.14	-0.21	0.07	-0.17	0.02	-0.05	0.12				
Arsenic (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			
Calcium (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		
Scandium (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	
Silicon (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 14-43: 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)
Platinum (ppm)	0.06														
Palladium (ppb)	0.41	0.43													
Nickel (pct)	-0.12	0.00	-0.07												
Cobalt (pct)	-0.02	-0.01	0.10	0.36											
Magnesium (pct)	-0.05	0.00	-0.05	0.01	-0.07										
Iron (pct)	-0.05	0.02	-0.07	0.24	0.06	-0.31									
Manganese (ppm)	-0.03	0.00	0.10	0.28	0.78	-0.07	0.09								
Zinc (ppm)	-0.06	0.00	-0.02	0.57	0.46	-0.11	0.40	0.35							
Copper (ppm)	0.19	-0.02	0.41	-0.11	0.08	-0.01	0.01	0.10	0.05						
Aluminium (pct)	0.13	0.00	0.21	-0.22	0.09	-0.04	-0.19	0.04	-0.23	0.20					
Chromium (ppm)	0.01	0.10	0.23	0.08	-0.16	-0.09	0.01	-0.20	-0.01	-0.06	0.04				
Arsenic (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			
Calcium (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		
Scandium (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	
Silicon (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 14-44: 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)
Platinum (ppm)	0.00														
Palladium (ppb)	0.53	0.12													
Nickel (pct)	0.00	0.09	-0.07												
Cobalt (pct)	0.09	0.06	0.17	0.41											
Magnesium (pct)	-0.03	0.01	0.05	0.11	-0.08										
Iron (pct)	0.09	0.12	0.11	0.61	0.45	-0.13									
Manganese (ppm)	0.05	0.07	0.16	0.39	0.72	-0.05	0.43								
Zinc (ppm)	0.14	0.08	0.18	0.66	0.53	-0.05	0.71	0.42							
Copper (ppm)	0.30	-0.01	0.39	-0.08	0.09	-0.01	0.11	0.09	0.18						
Aluminium (pct)	0.30	0.02	0.35	-0.04	0.19	-0.02	0.22	0.13	0.19	0.27					
Chromium (ppm)	-0.04	0.15	-0.10	0.45	0.13	-0.07	0.44	0.15	0.37	-0.04	0.01				
Arsenic (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			
Calcium (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		
Scandium (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	
Silicon (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 14-45: Correlation Matrix SGZ

14.1.7 Variography

Variography has been carried out for the GZ, SGZ and TZ zones for cobalt, nickel, iron, silicon, aluminium, magnesium, manganese, platinum and scandium. 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 14-6 to Table 14-8 below and Figure 14-46 to Figure 14-72.

Table 14-6: GZ Variogram Model Parameters

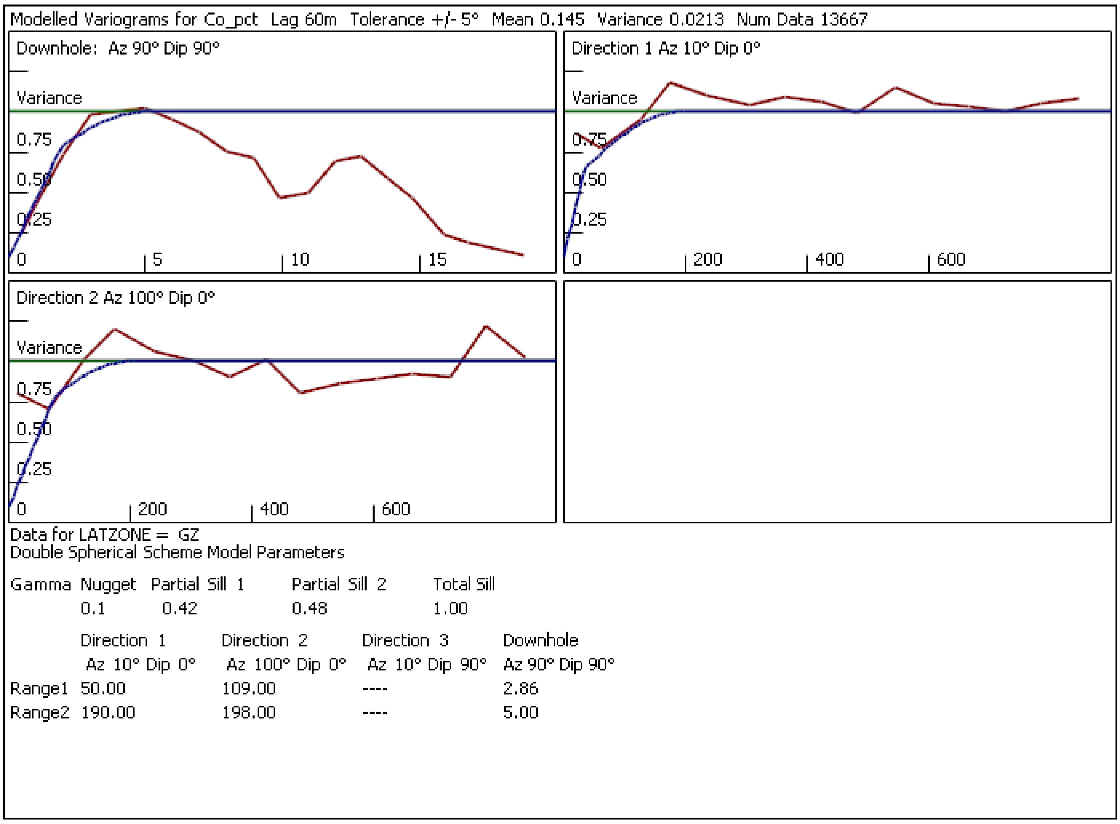
LATZONE GZ Variable	Variances			Major Axis (East-West)		Semi-Major Axis (North-South)		Minor Axis (Vertical)		Rotation
	Nugget	Sill 1	Sill 2	Range 1	Range 2	Range 1	Range 2	Range 1	Range 2	
Nickel (%)	0.10	0.32	0.68	138.0	465.0	130.0	420.0	8.6	12.7	45.0
Cobalt (%)	0.10	0.42	0.48	109.0	198.0	50.0	190.0	2.9	5.0	10.0
Iron (%)	0.12	0.42	0.31	89.0	225.0	50.0	230.0	4.6	15.8	0.0
Silicon (%)	0.08	0.44	0.37	119.0	257.0	100.0	225.0	7.7	7.7	0.0
Aluminium (%)	0.07	0.37	0.56	89.0	366.0	40.0	460.0	8.6	9.2	0.0
Magnesium (%)	0.08	0.21	0.71	59.0	267.0	128.6	320.0	2.0	8.1	0.0
Manganese (ppm)	0.30	0.50	0.20	128.6	225.0	80.0	225.0	4.0	6.8	5.0
Platinum (ppm)	0.07	0.73	0.20	175.0	212.0	187.0	210.0	1.1	2.6	0.0
Scandium (ppm)	0.08	0.17	0.75	40.0	180.0	112.5	180.0	11.2	11.2	0.0

Table 14-7: SGZ Variogram Model Parameters

LAT ZONE SGZ Variable	Variances			Major Axis (East-West)		Semi-Major Axis (North-South)		Minor Axis (Vertical)		Rotation
	Nugget	Sill 1	Sill 2	Range 1	Range 2	Range 1	Range 2	Range 1	Range 2	
Nickel (%)	0.13	0.38	0.43	109.0	225.0	160.0	250.0	7.3	10.3	0.0
Cobalt (%)	0.12	0.30	0.58	128.6	225.0	100.0	370.0	2.6	6.6	10.0
Iron (%)	0.12	0.37	0.43	89.0	225.0	80.0	225.0	3.7	5.5	0.0
Silicon (%)	0.08	0.36	0.46	128.6	158.0	40.0	160.0	2.9	5.0	0.0
Aluminium (%)	0.03	0.46	0.38	148.0	702.0	170.0	800.0	7.5	10.8	0.0
Magnesium (%)	0.19	0.11	0.70	128.6	225.0	200.0	225.0	3.5	7.3	0.0
Manganese (ppm)	0.19	0.47	0.34	326.0	326.0	110.0	380.0	4.6	8.4	5.0
Platinum (ppm)	0.29	0.51	0.20	128.6	225.0	90.0	310.0	8.4	8.4	0.0
Scandium (ppm)	0.13	0.04	0.83	99.0	257.0	128.6	225.0	5.5	16.7	0.0

Table 14-8: TZ Variogram Model Parameters

LAT ZONE TZ Variable	Variances			Major Axis (East-West)		Semi-Major Axis (North-South)		Minor Axis (Vertical)		Rotation
	Nugget	Sill 1	Sill 2	Range 1	Range 2	Range 1	Range 2	Range 1	Range 2	
Nickel (%)	0.02	0.32	0.66	40.0	225.0	20.0	225.0	2.9	5.0	0.0
Cobalt (%)	0.10	0.70	0.20	128.6	225.0	50.0	225.0	1.8	5.0	0.0
Iron (%)	0.05	0.07	0.77	49.0	188.0	60.0	225.0	2.9	6.6	0.0
Silicon (%)	0.24	0.06	0.70	128.6	267.0	110.0	290.0	5.3	10.3	0.0
Aluminium (%)	0.15	0.47	0.38	79.0	225.0	80.0	225.0	4.4	5.0	0.0
Magnesium (%)	0.10	0.00	0.90	10.0	178.0	250.0	330.0	2.9	5.0	0.0
Manganese (ppm)	0.19	0.61	0.20	227.0	316.0	60.0	225.0	2.9	5.0	5.0
Platinum (ppm)	0.10	0.07	0.38	128.6	225.0	20.0	350.0	0.4	3.7	0.0
Scandium (ppm)	0.07	0.55	0.38	69.0	237.0	100.0	310.0	5.3	7.7	0.0



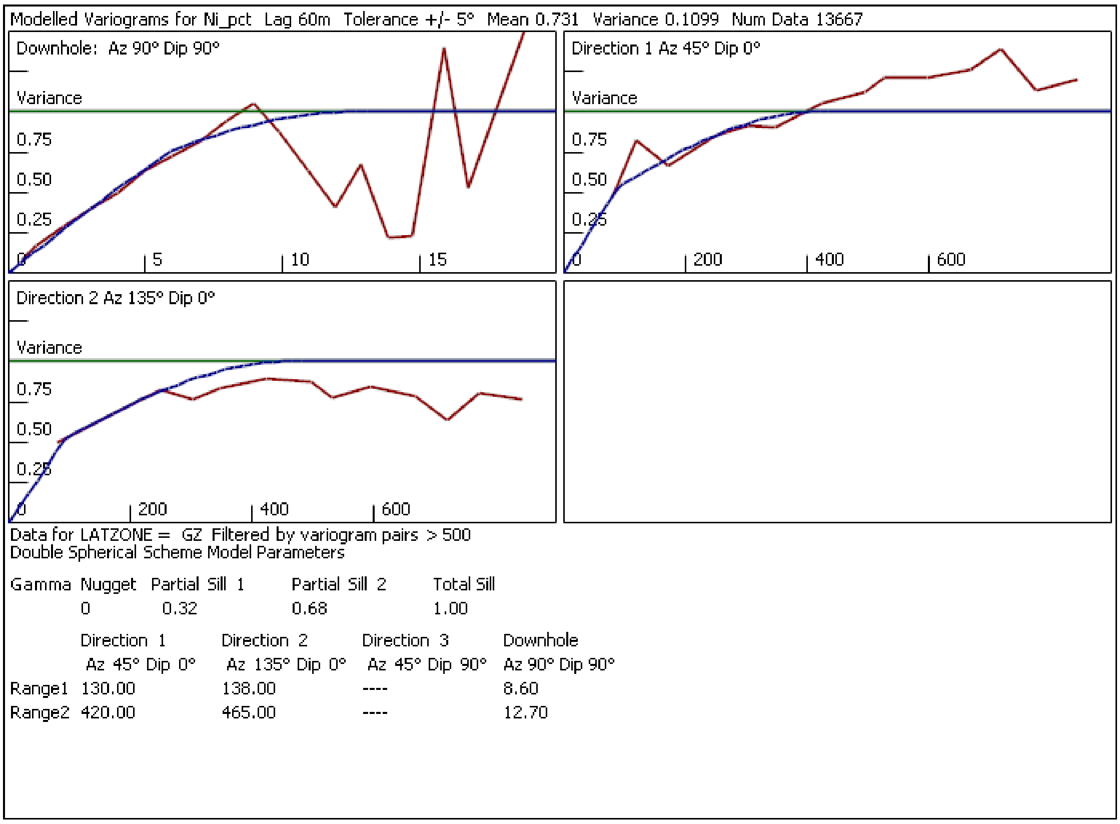
Modelled Variograms for Co_pct Lag 60m Tolerance +/- 5° Mean 0.145 Variance 0.0213 Num Data 13667

Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.1	0.42	0.48	1.00

	Direction 1 Az 10° Dip 0°	Direction 2 Az 100° Dip 0°	Direction 3 Az 10° Dip 90°	Downhole Az 90° Dip 90°
Range1	50.00	109.00	----	2.86
Range2	190.00	198.00	----	5.00

Figure 14-46: GZ Co



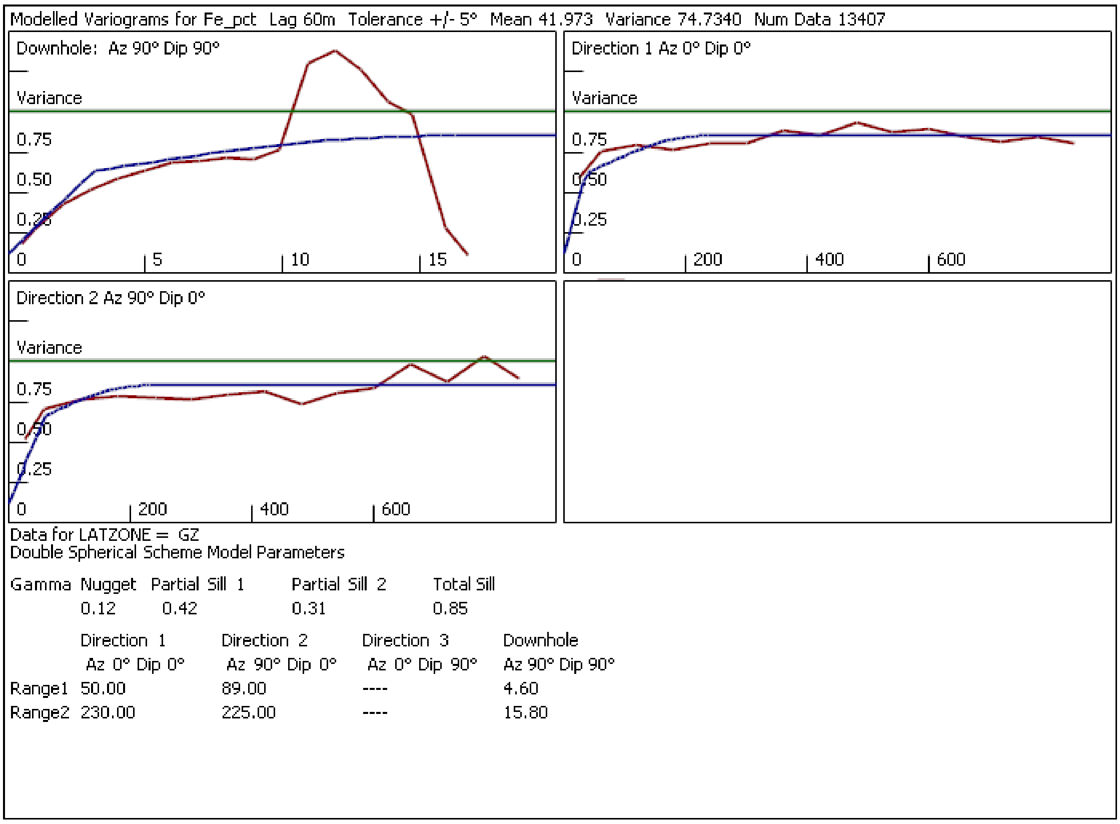
Modelled Variograms for Ni_pct Lag 60m Tolerance +/- 5° Mean 0.731 Variance 0.1099 Num Data 13667

Data for LATZONE = GZ Filtered by variogram pairs > 500
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0	0.32	0.68	1.00

	Direction 1 Az 45° Dip 0°	Direction 2 Az 135° Dip 0°	Direction 3 Az 45° Dip 90°	Downhole Az 90° Dip 90°
Range1	130.00	138.00	----	8.60
Range2	420.00	465.00	----	12.70

Figure 14-47: GZ Ni



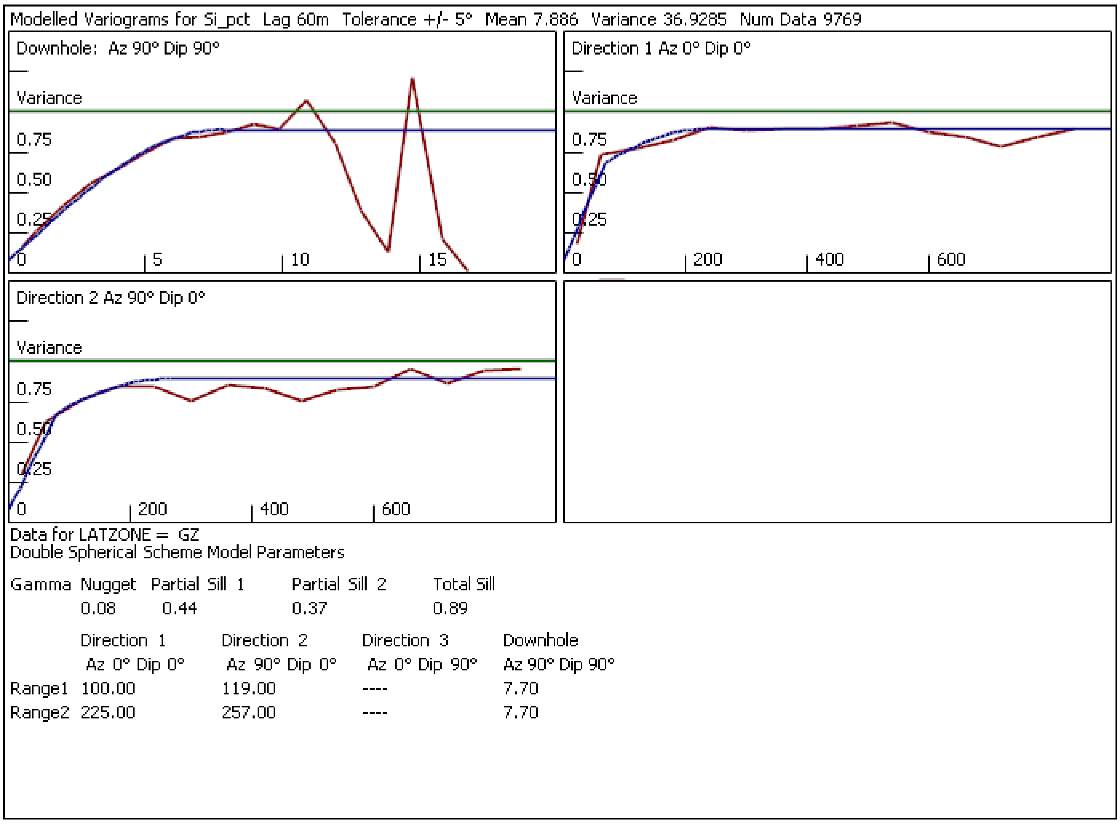
Modelled Variograms for Fe_pct Lag 60m Tolerance +/- 5° Mean 41.973 Variance 74.7340 Num Data 13407

```
Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2      Total Sill
       0.12     0.42               0.31                0.85

       Direction 1       Direction 2       Direction 3       Downhole
        Az 0° Dip 0°      Az 90° Dip 0°     Az 0° Dip 90°    Az 90° Dip 90°
Range1  50.00            89.00             ----              4.60
Range2  230.00           225.00            ----              15.80
```

Figure 14-48: GZ Fe



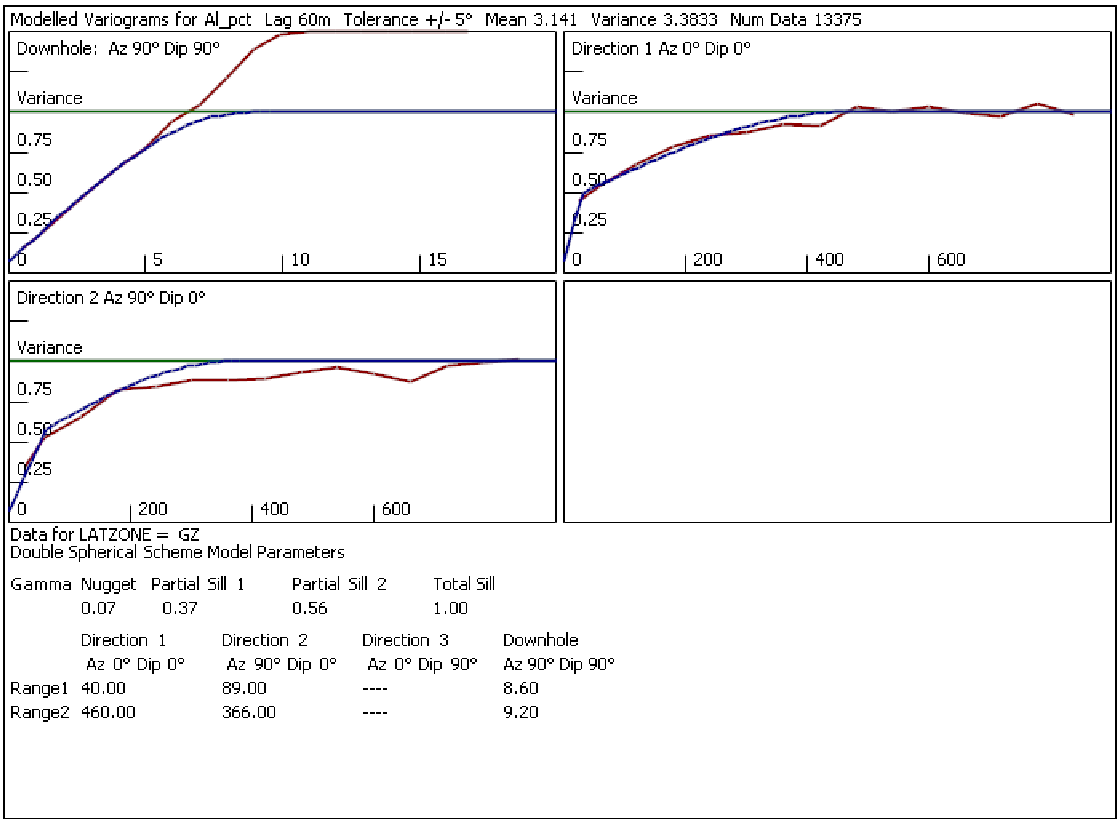
Modelled Variograms for Si_pct Lag 60m Tolerance +/- 5° Mean 7.886 Variance 36.9285 Num Data 9769

```
Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2      Total Sill
       0.08     0.44               0.37                0.89

       Direction 1       Direction 2       Direction 3       Downhole
        Az 0° Dip 0°      Az 90° Dip 0°     Az 0° Dip 90°    Az 90° Dip 90°
Range1  100.00           119.00            ----              7.70
Range2  225.00           257.00            ----              7.70
```

Figure 14-49: GZ Si



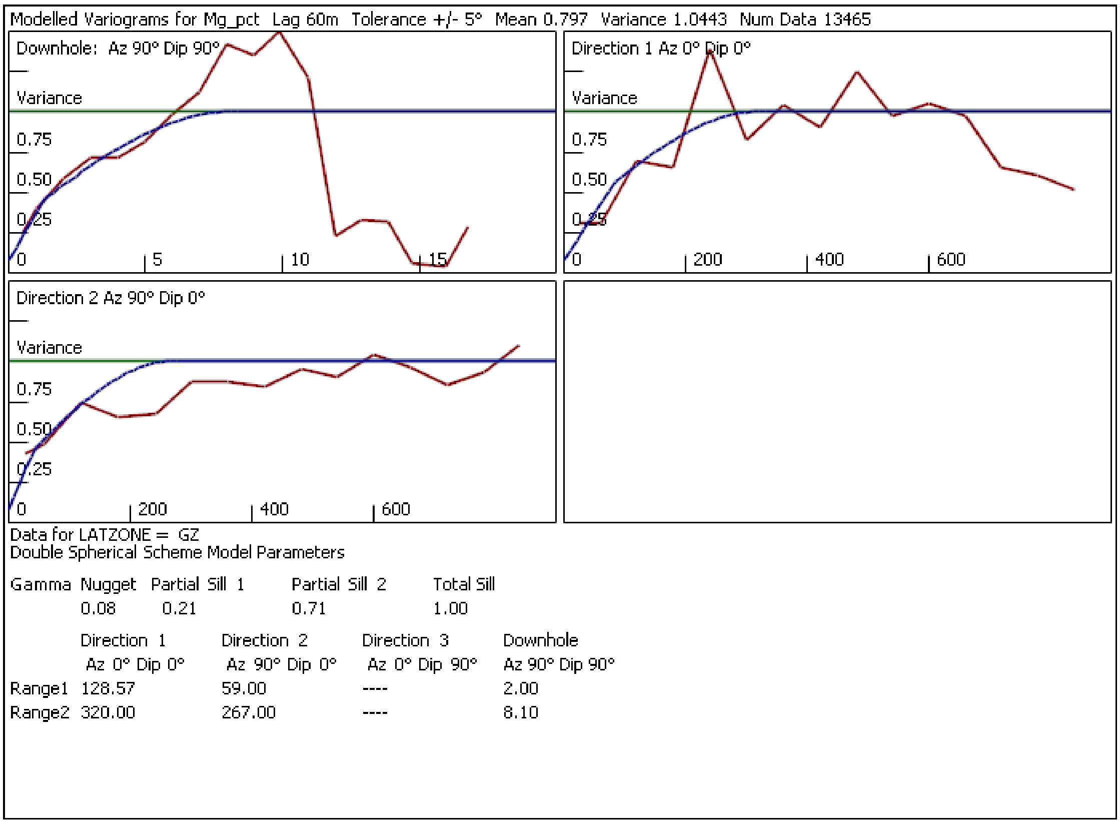
Modelled Variograms for Al_pct Lag 60m Tolerance +/- 5° Mean 3.141 Variance 3.3833 Num Data 13375

Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.07	0.37	0.56	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	40.00	89.00	----	8.60
Range2	460.00	366.00	----	9.20

Figure 14-50: GZ Al



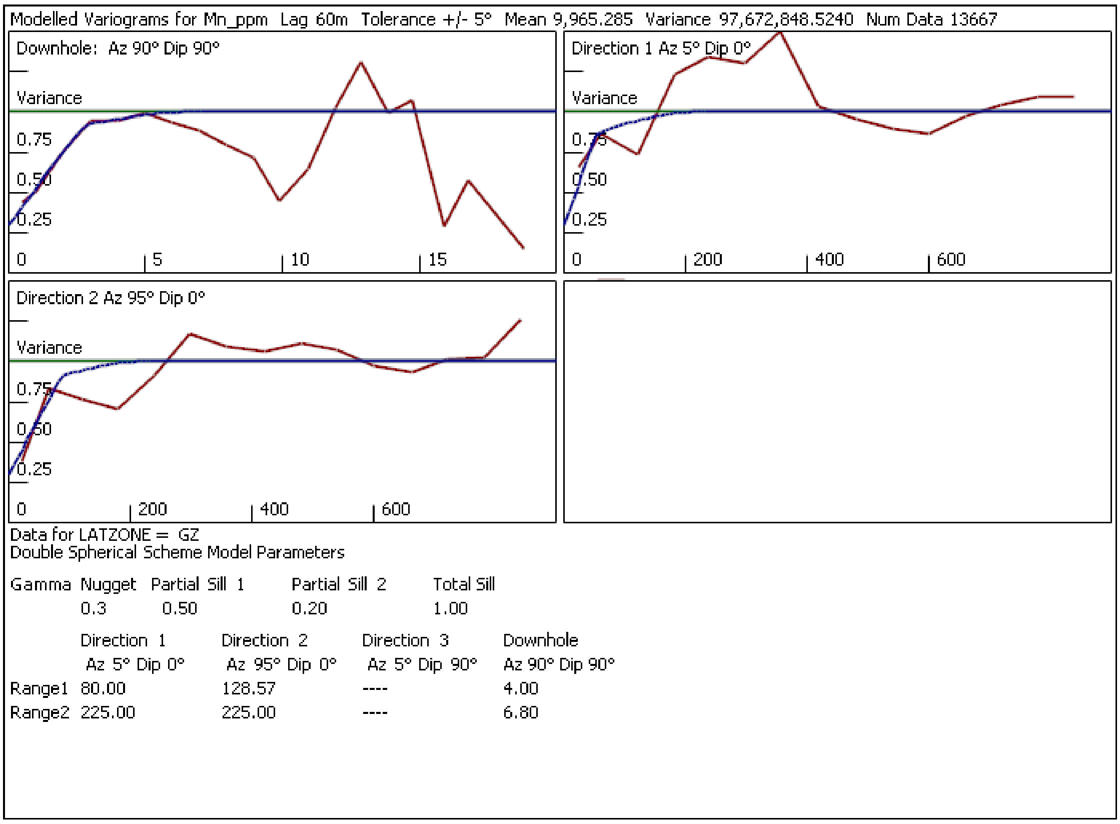
Modelled Variograms for Mg_pct Lag 60m Tolerance +/- 5° Mean 0.797 Variance 1.0443 Num Data 13465

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 14-51: GZ Mg



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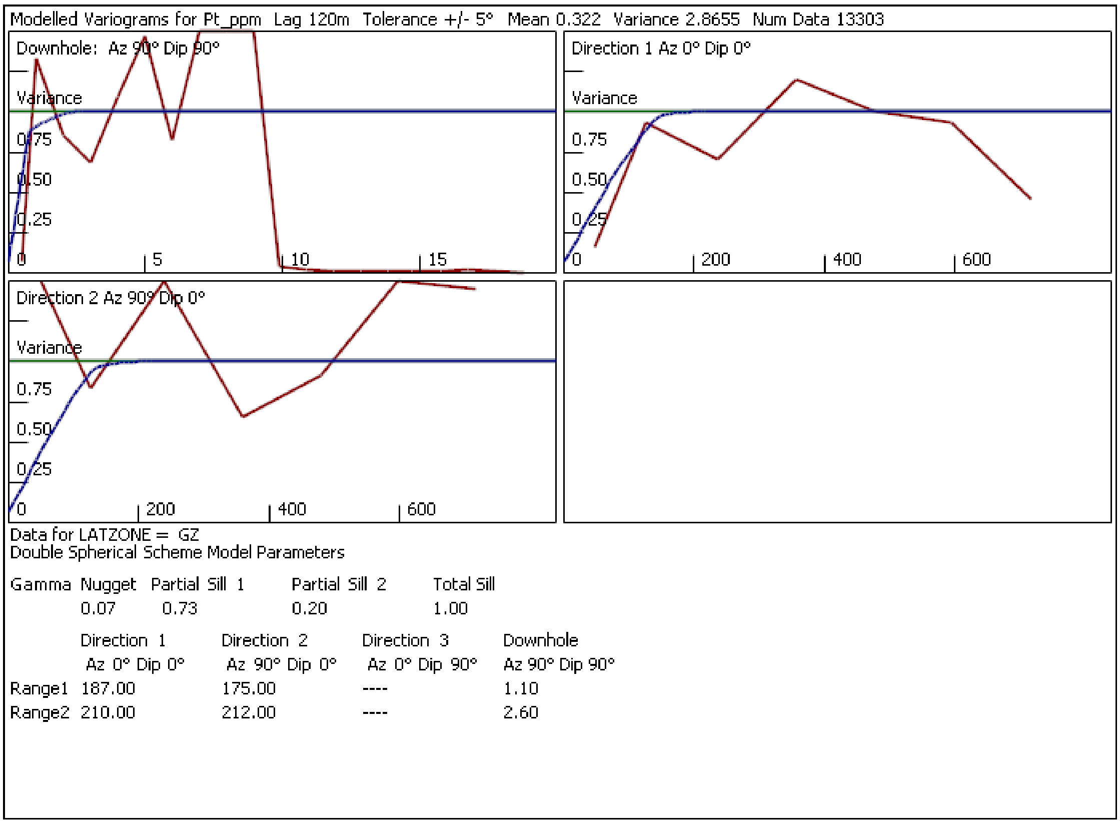
Modelled Variograms for Mn_ppm  Lag 60m  Tolerance +/- 5°  Mean 9,965.285  Variance 97,672,848.5240  Num Data 13667
```

```
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       Direction 2       Direction 3       Downhole
        Az 5° Dip 0°      Az 95° Dip 0°     Az 5° Dip 90°    Az 90° Dip 90°
Range1  80.00            128.57            ----              4.00
Range2  225.00           225.00            ----              6.80
```

Figure 14-52: GZ Mn



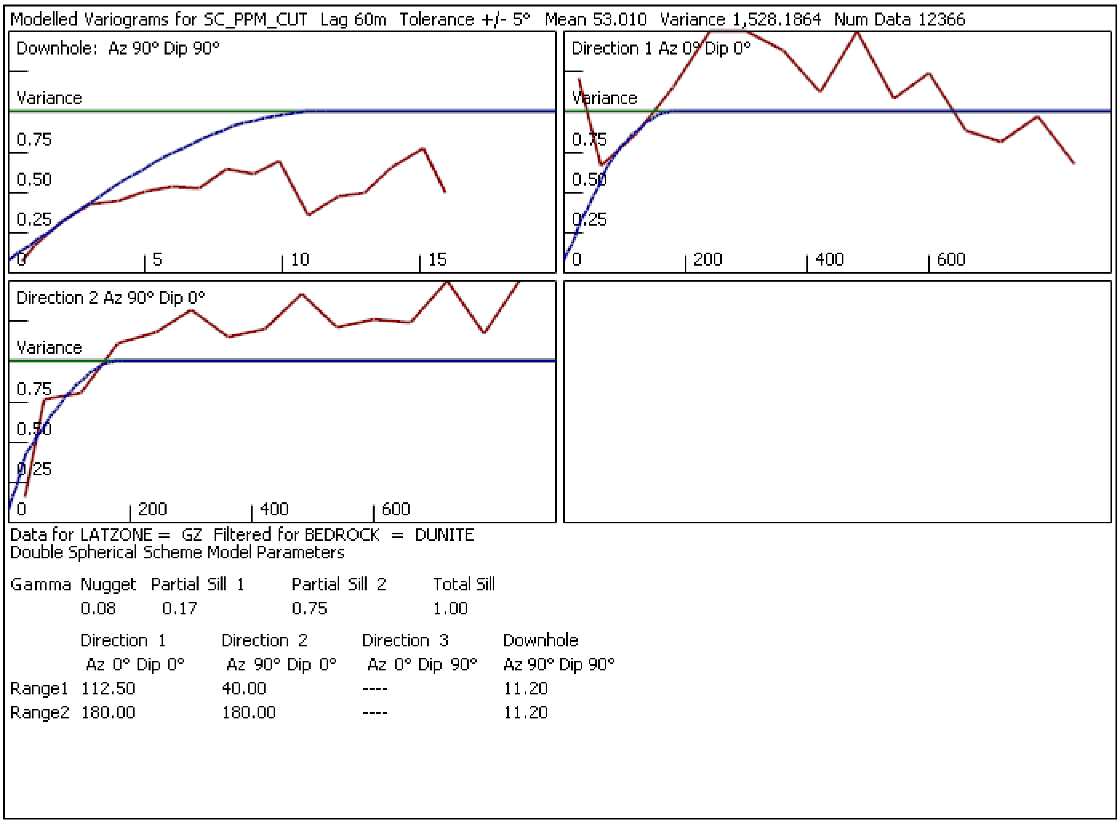
```
Modelled Variograms for Pt_ppm  Lag 120m  Tolerance +/- 5°  Mean 0.322  Variance 2.8655  Num Data 13303
```

```
Data for LATZONE = GZ
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1    Partial Sill 2    Total Sill
       0.07    0.73              0.20              1.00

       Direction 1       Direction 2       Direction 3       Downhole
        Az 0° Dip 0°      Az 90° Dip 0°     Az 0° Dip 90°    Az 90° Dip 90°
Range1  187.00           175.00            ----              1.10
Range2  210.00           212.00            ----              2.60
```

Figure 14-53: GZ Pt



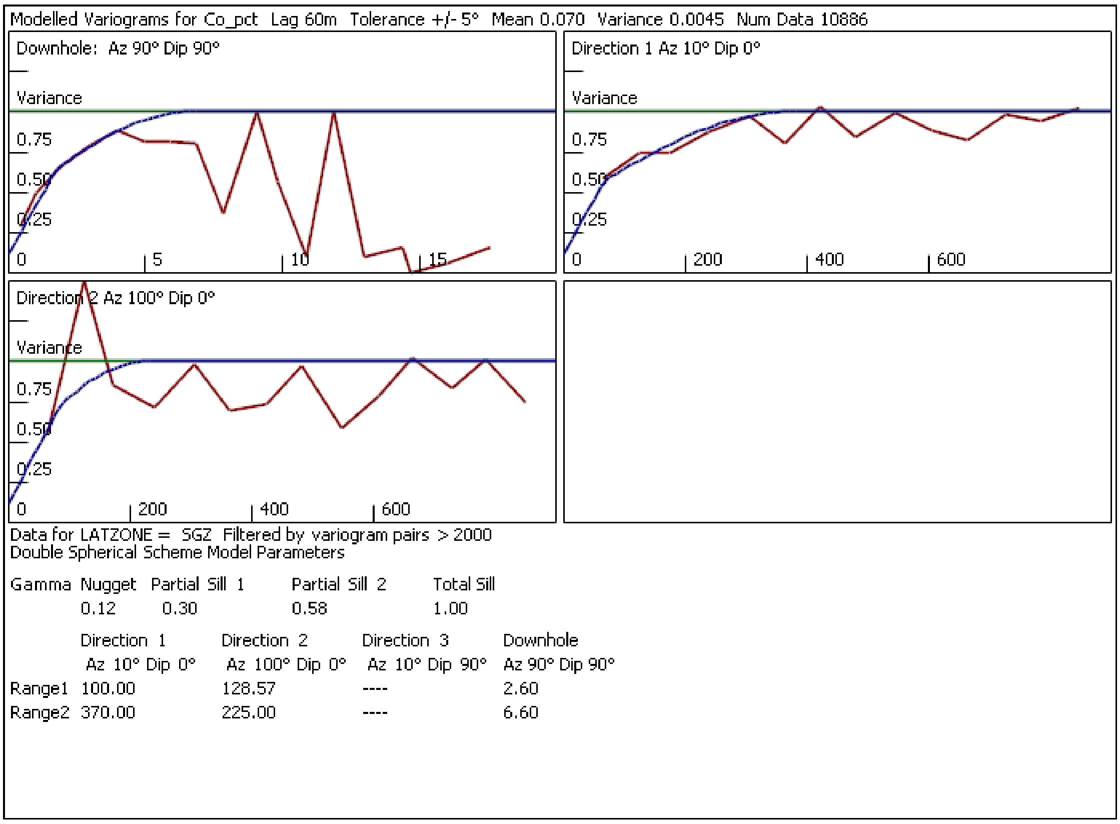
Modelled Variograms for SC_PPM_CUT Lag 60m Tolerance +/- 5° Mean 53.010 Variance 1,528.1864 Num Data 12366

```
Data for LATZONE =  GZ  Filtered for BEDROCK  =  DUNITE
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill  1      Partial Sill  2      Total Sill
       0.08       0.17              0.75                 1.00

         Direction  1       Direction  2       Direction  3       Downhole
          Az  0° Dip 0°      Az  90° Dip  0°    Az  0° Dip  90°   Az 90° Dip 90°
Range1  112.50             40.00              ----               11.20
Range2  180.00             180.00             ----               11.20
```

Figure 14-54: GZ Sc



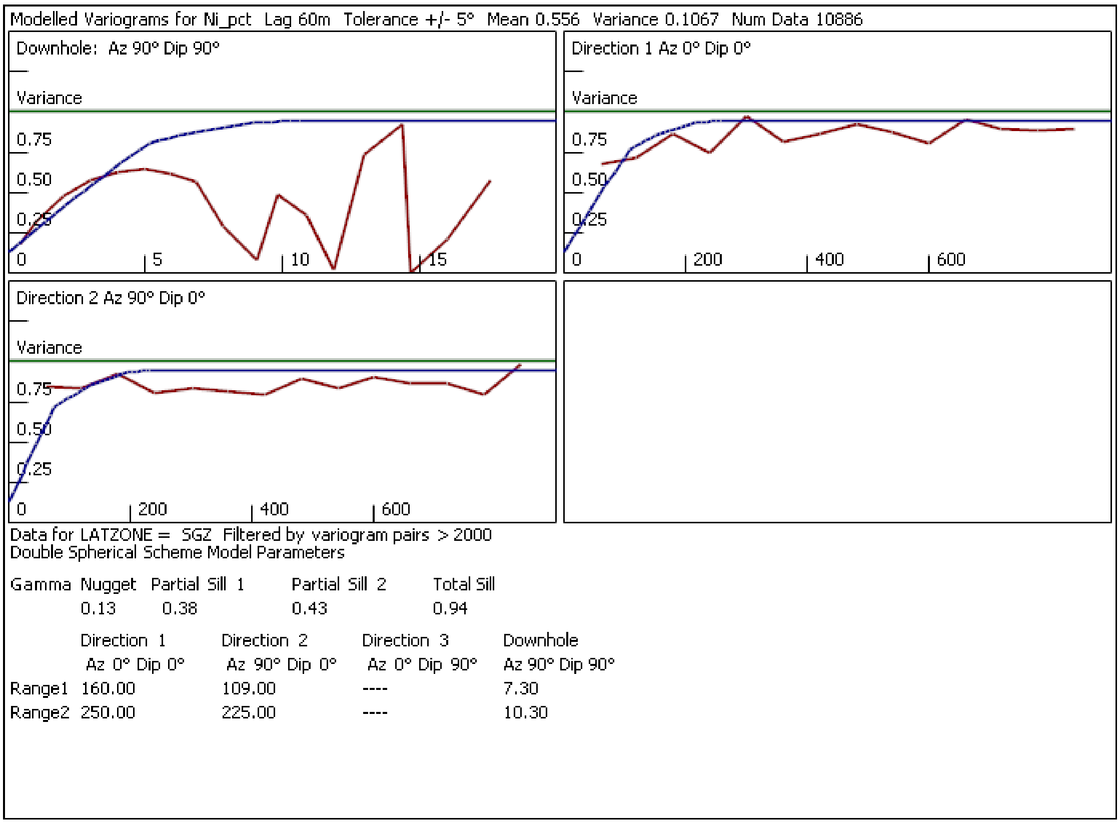
Modelled Variograms for Co_pct Lag 60m Tolerance +/- 5° Mean 0.070 Variance 0.0045 Num Data 10886

```
Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill  1      Partial Sill  2      Total Sill
       0.12       0.30              0.58                 1.00

         Direction  1       Direction  2       Direction  3       Downhole
          Az  10° Dip  0°    Az  100° Dip  0°   Az  10° Dip  90°  Az 90° Dip 90°
Range1  100.00             128.57             ----               2.60
Range2  370.00             225.00             ----               6.60
```

Figure 14-55: SGZ Co



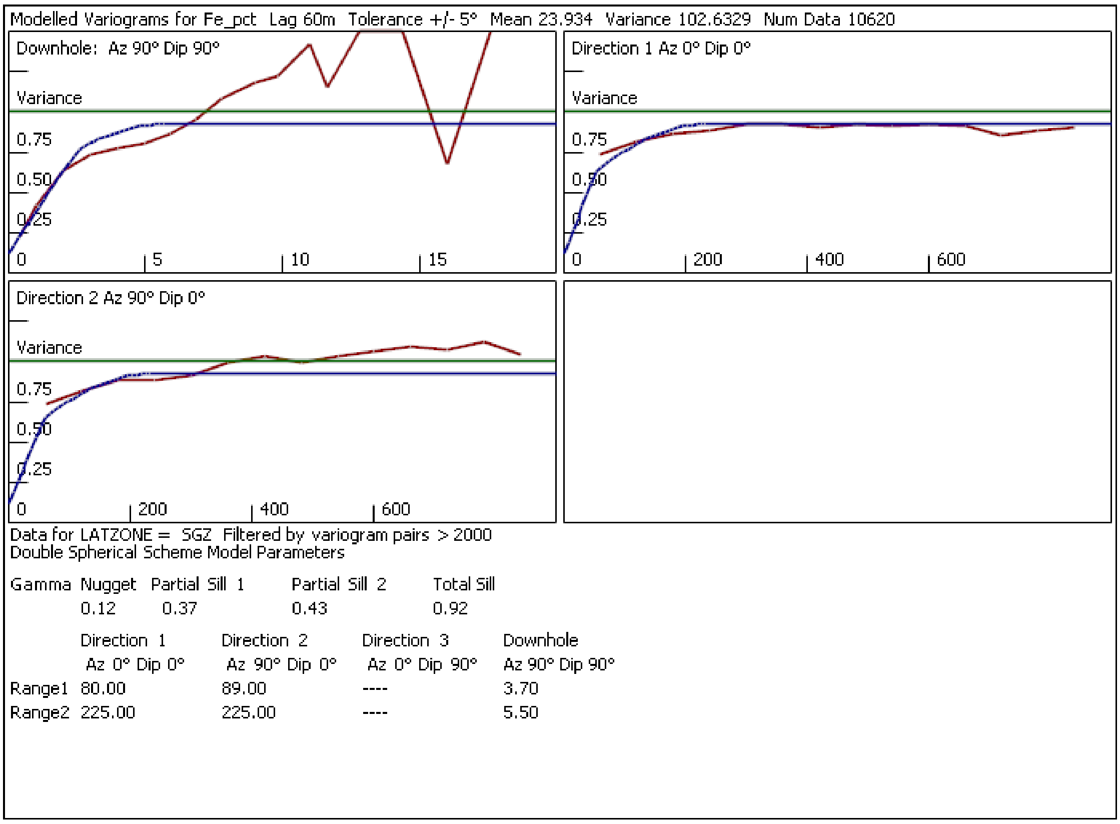
```
Modelled Variograms for Ni_pct  Lag 60m  Tolerance +/- 5°  Mean 0.556  Variance 0.1067  Num Data 10886

Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2     Total Sill
       0.13      0.38              0.43               0.94

       Direction 1       Direction 2      Direction 3       Downhole
        Az 0° Dip 0°      Az 90° Dip 0°    Az 0° Dip 90°    Az 90° Dip 90°
Range1  160.00           109.00           ----              7.30
Range2  250.00           225.00           ----              10.30
```

Figure 14-56: SGZ Ni



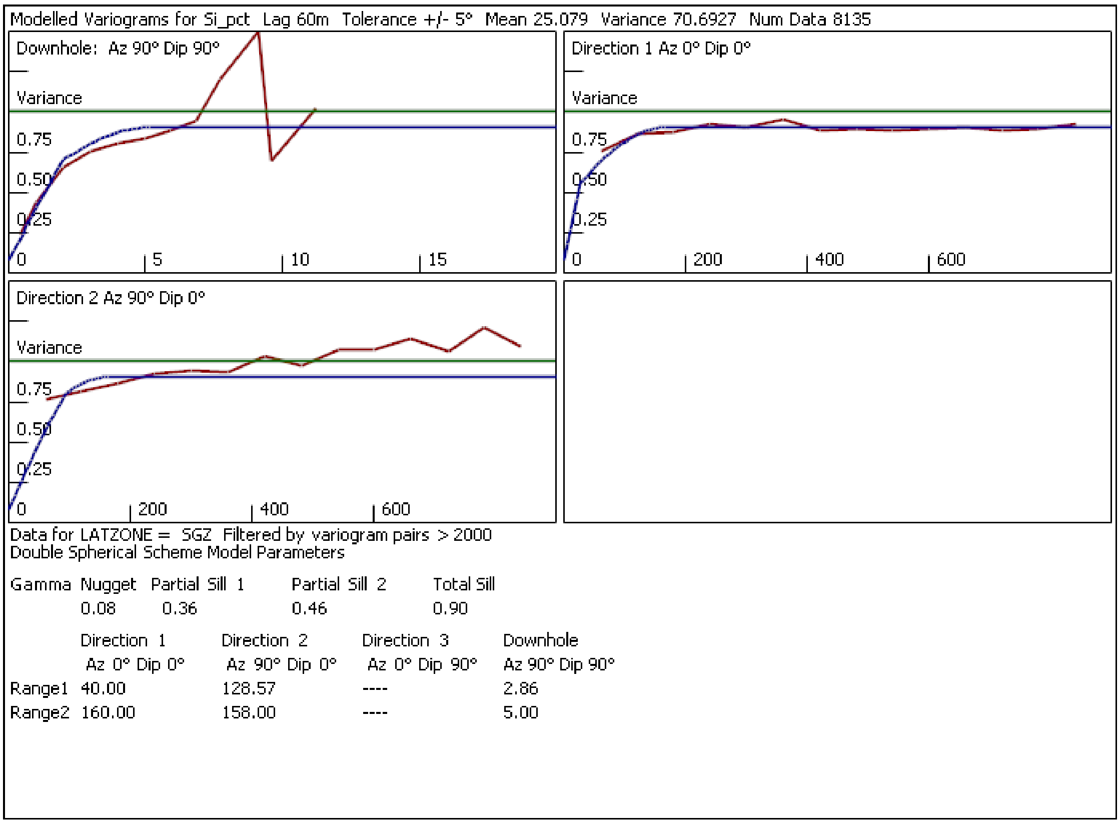
```
Modelled Variograms for Fe_pct  Lag 60m  Tolerance +/- 5°  Mean 23.934  Variance 102.6329  Num Data 10620

Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2     Total Sill
       0.12      0.37              0.43               0.92

       Direction 1       Direction 2      Direction 3       Downhole
        Az 0° Dip 0°      Az 90° Dip 0°    Az 0° Dip 90°    Az 90° Dip 90°
Range1  80.00            89.00            ----              3.70
Range2  225.00           225.00           ----              5.50
```

Figure 14-57: SGZ Fe

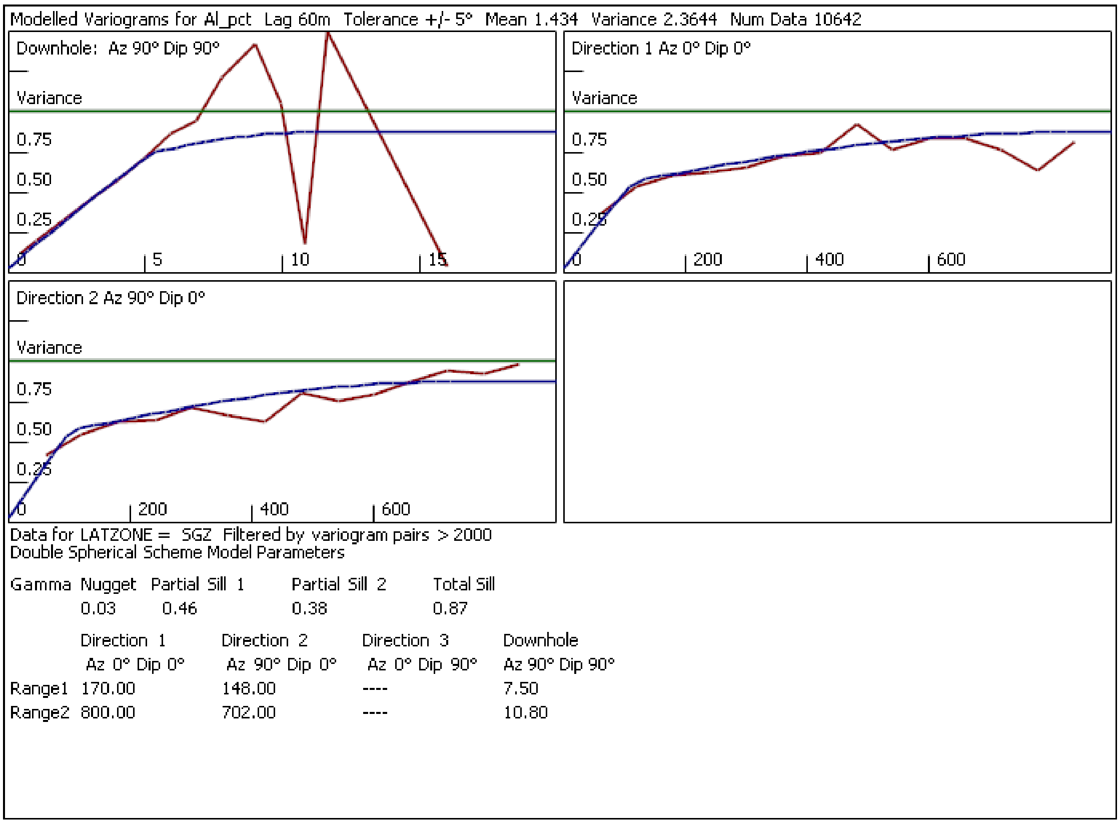


```
Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill 1       Partial Sill 2       Total Sill
       0.08     0.36                 0.46                 0.90

       Direction 1      Direction 2      Direction 3      Downhole
        Az 0° Dip 0°     Az 90° Dip 0°    Az 0° Dip 90°   Az 90° Dip 90°
Range1  40.00           128.57           ----             2.86
Range2  160.00          158.00           ----             5.00
```

Figure 14-58: SGZ Si

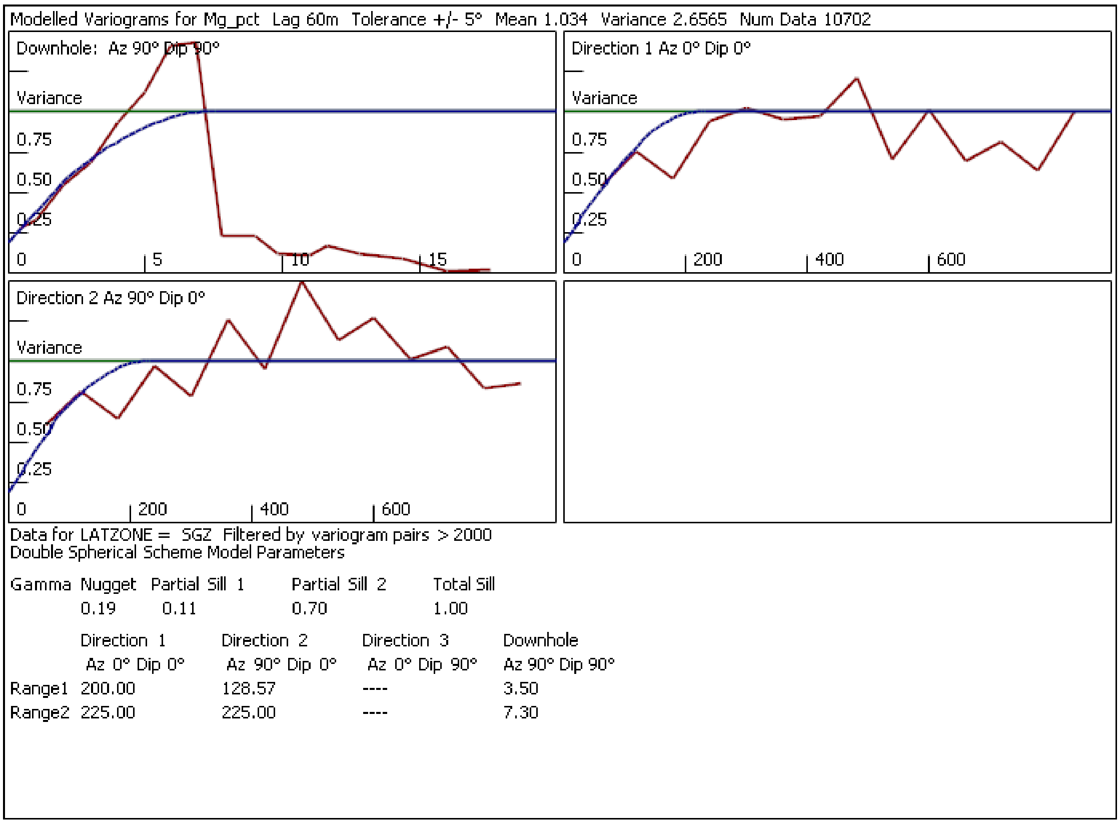


```
Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill 1       Partial Sill 2       Total Sill
       0.03     0.46                 0.38                 0.87

       Direction 1      Direction 2      Direction 3      Downhole
        Az 0° Dip 0°     Az 90° Dip 0°    Az 0° Dip 90°   Az 90° Dip 90°
Range1  170.00          148.00           ----             7.50
Range2  800.00          702.00           ----             10.80
```

Figure 14-59: SGZ Al

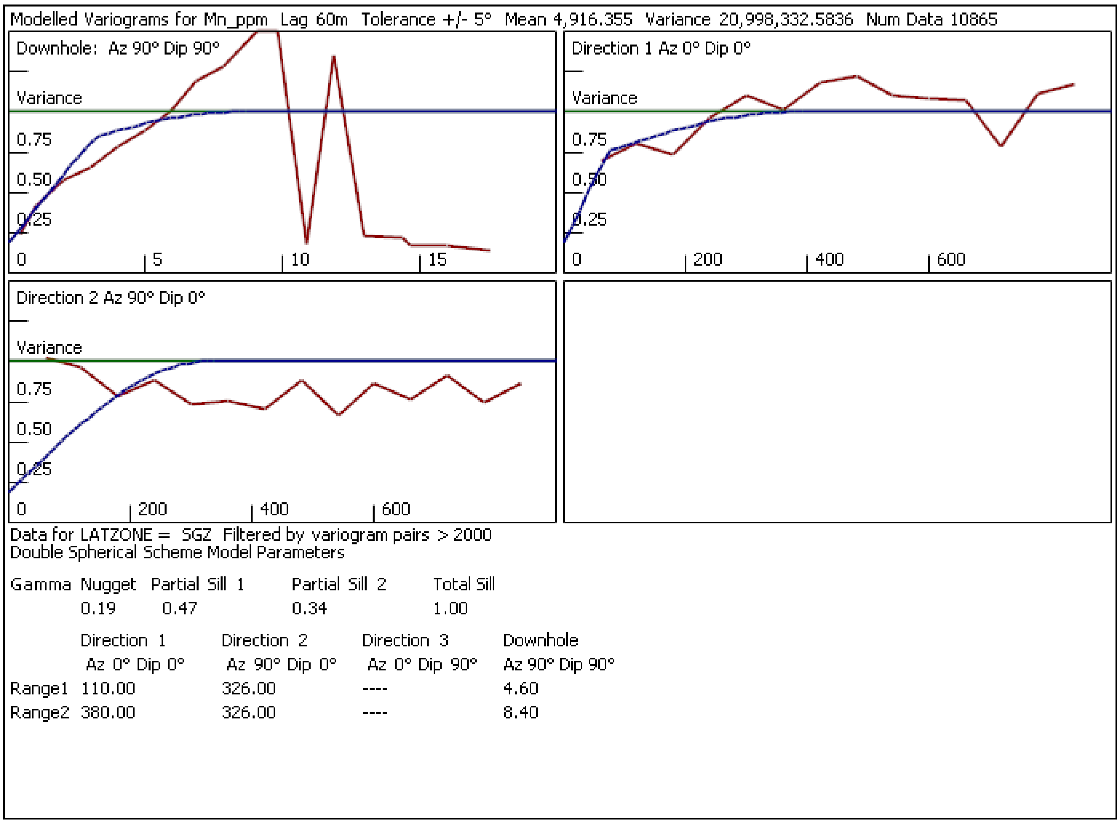

Modelled Variograms for Mg_pct Lag 60m Tolerance +/- 5° Mean 1.034 Variance 2.6565 Num Data 10702

Data for LATZONE = SGZ Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.19	0.11	0.70	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	200.00	128.57	----	3.50
Range2	225.00	225.00	----	7.30

Figure 14-60: SGZ Mg

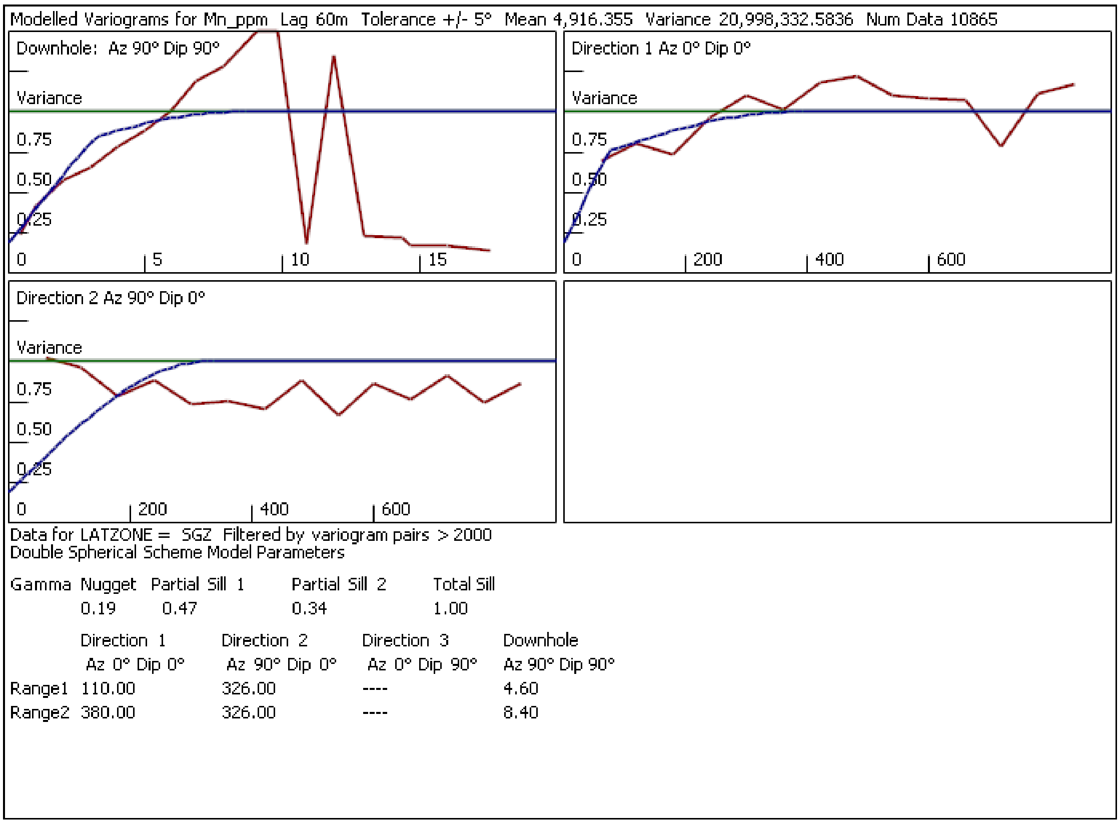

Modelled Variograms for Mn_ppm Lag 60m Tolerance +/- 5° Mean 4,916.355 Variance 20,998,332.5836 Num Data 10865

Data for LATZONE = SGZ Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.19	0.47	0.34	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	110.00	326.00	----	4.60
Range2	380.00	326.00	----	8.40

Figure 14-61: SGZ Mn



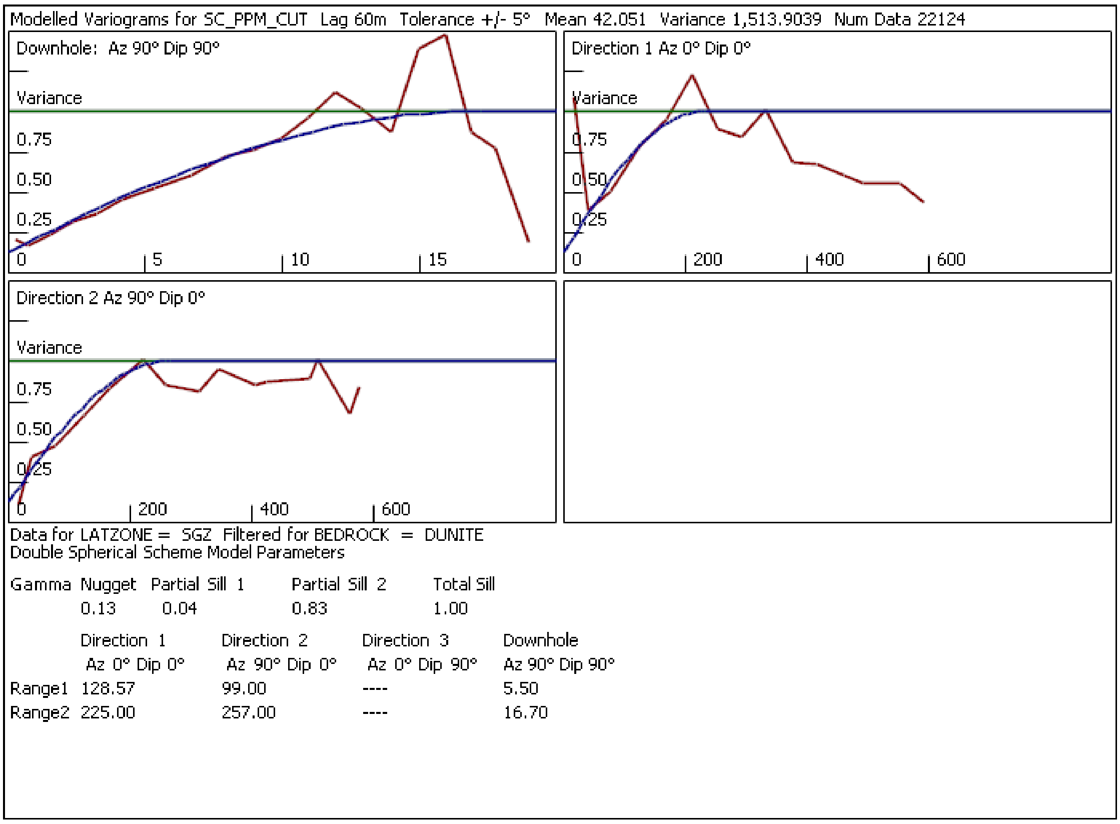
```
Modelled Variograms for Mn_ppm  Lag 60m  Tolerance +/- 5°  Mean 4,916.355  Variance 20,998,332.5836  Num Data 10865
```

```
Data for LATZONE =  SGZ  Filtered by variogram pairs > 2000
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill  1      Partial Sill  2      Total Sill
        0.19       0.47              0.34                 1.00

        Direction  1         Direction  2        Direction  3        Downhole
         Az  0° Dip 0°        Az  90° Dip  0°     Az  0° Dip  90°     Az 90° Dip 90°
Range1  110.00               326.00              ----                4.60
Range2  380.00               326.00              ----                8.40
```

Figure 14-62: SGZ Pt



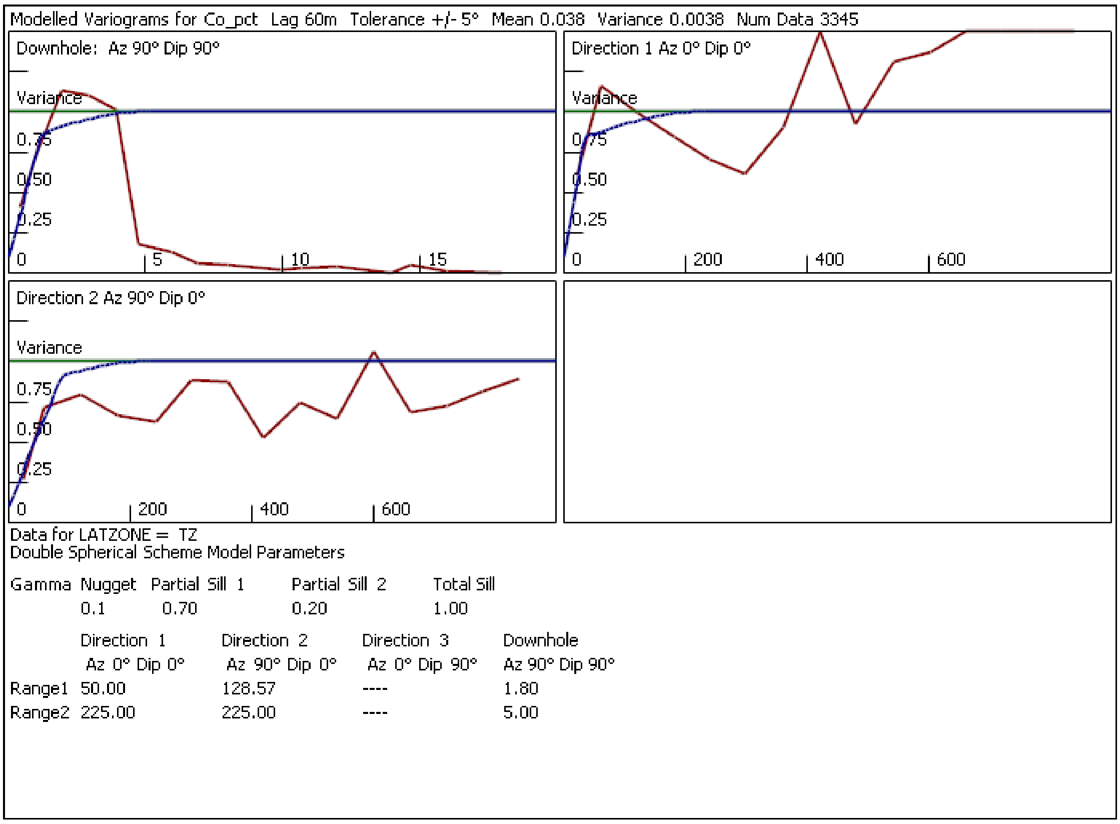
```
Modelled Variograms for SC_PPM_CUT  Lag 60m  Tolerance +/- 5°  Mean 42.051  Variance 1,513.9039  Num Data 22124
```

```
Data for LATZONE =  SGZ  Filtered for BEDROCK  =  DUNITE
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill  1      Partial Sill  2      Total Sill
        0.13       0.04              0.83                 1.00

        Direction  1         Direction  2        Direction  3        Downhole
         Az  0° Dip 0°        Az  90° Dip  0°     Az  0° Dip  90°     Az 90° Dip 90°
Range1  128.57               99.00               ----                5.50
Range2  225.00               257.00              ----                16.70
```

Figure 14-63: SGZ Sc



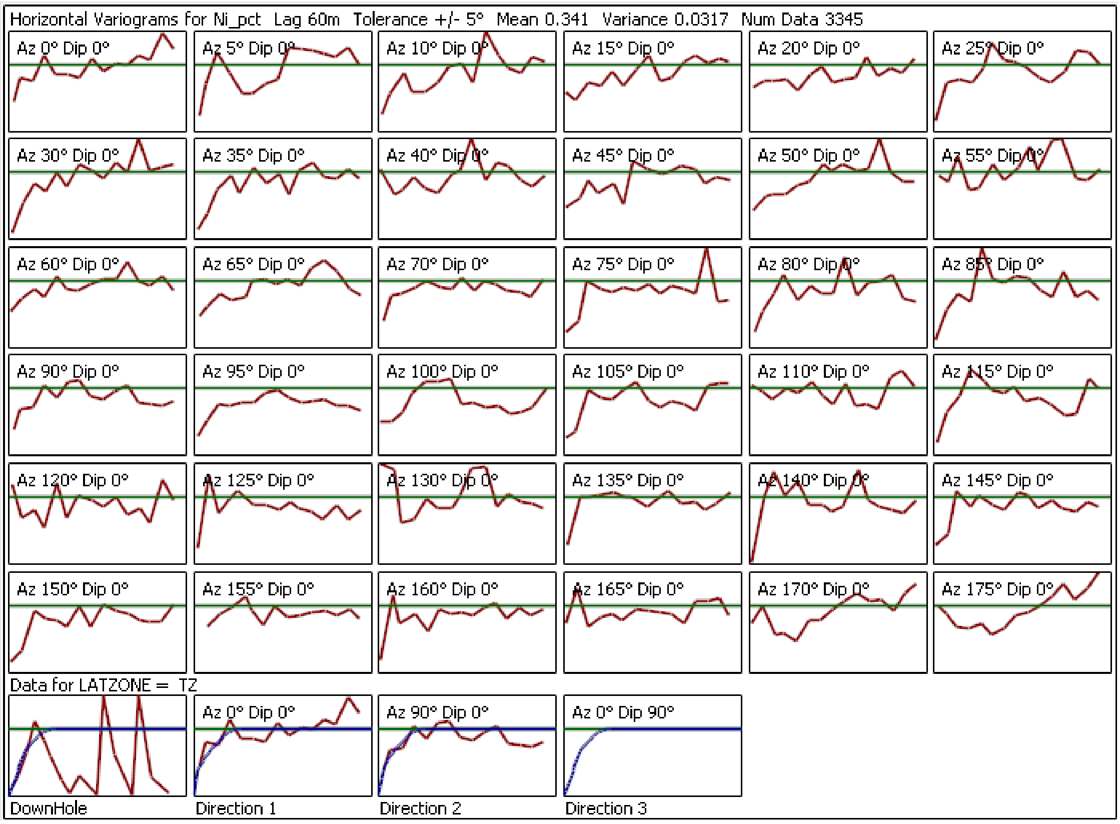
Modelled Variograms for Co_pct Lag 60m Tolerance +/- 5° Mean 0.038 Variance 0.0038 Num Data 3345

Data for LATZONE = TZ
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.1	0.70	0.20	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	50.00	128.57	----	1.80
Range2	225.00	225.00	----	5.00

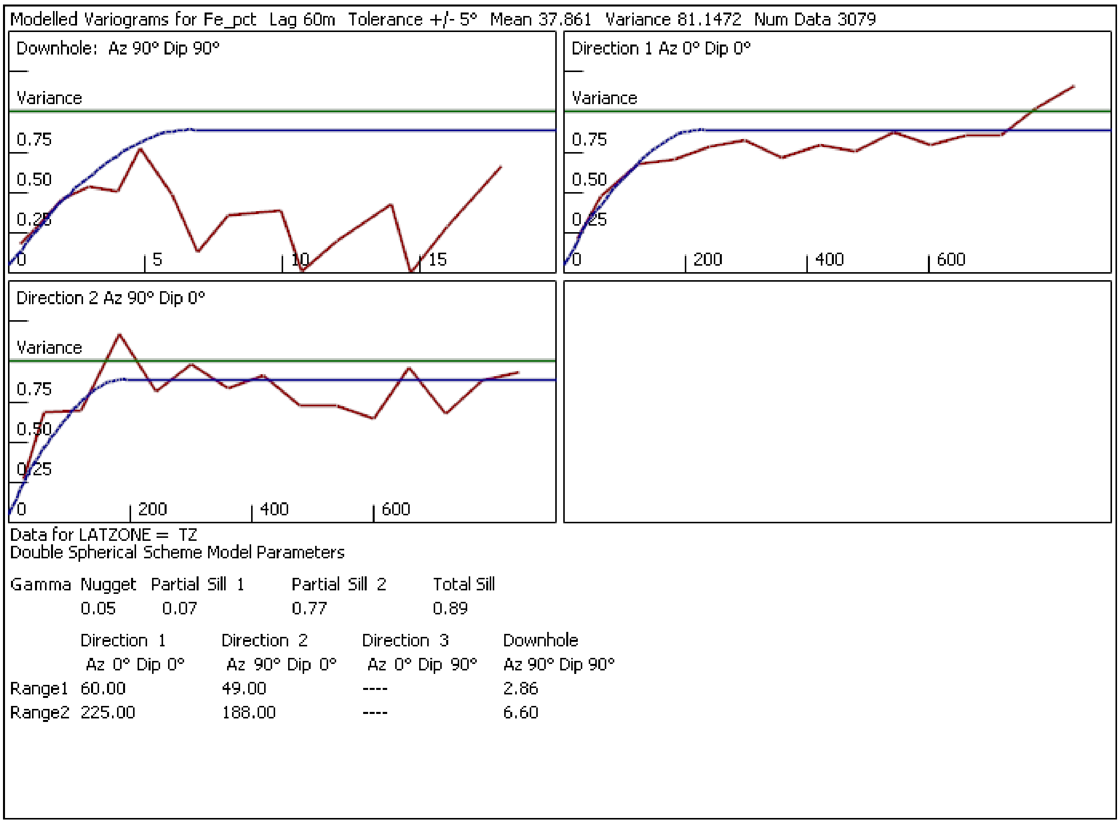
Figure 14-64: TZ Co



Horizontal Variograms for Ni_pct Lag 60m Tolerance +/- 5° Mean 0.341 Variance 0.0317 Num Data 3345

Data for LATZONE = TZ

Figure 14-65: TZ Ni



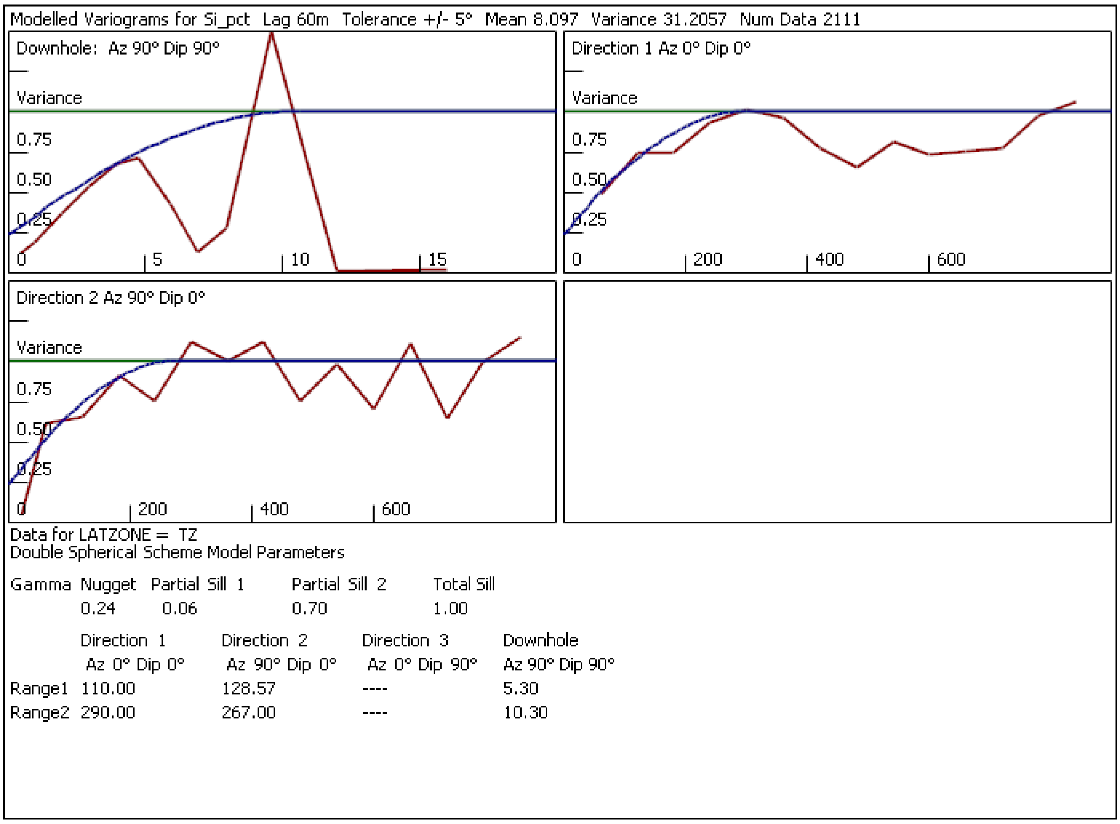
```
Modelled Variograms for Fe_pct  Lag 60m  Tolerance +/- 5°  Mean 37.861  Variance 81.1472  Num Data 3079

Data for LATZONE =  TZ
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill 1     Partial Sill 2     Total Sill
       0.05       0.07              0.77               0.89

       Direction 1      Direction 2      Direction 3      Downhole
        Az 0° Dip 0°     Az 90° Dip 0°    Az 0° Dip 90°   Az 90° Dip 90°
Range1  60.00           49.00            ----             2.86
Range2  225.00          188.00           ----             6.60
```

Figure 14-66: TZ Fe



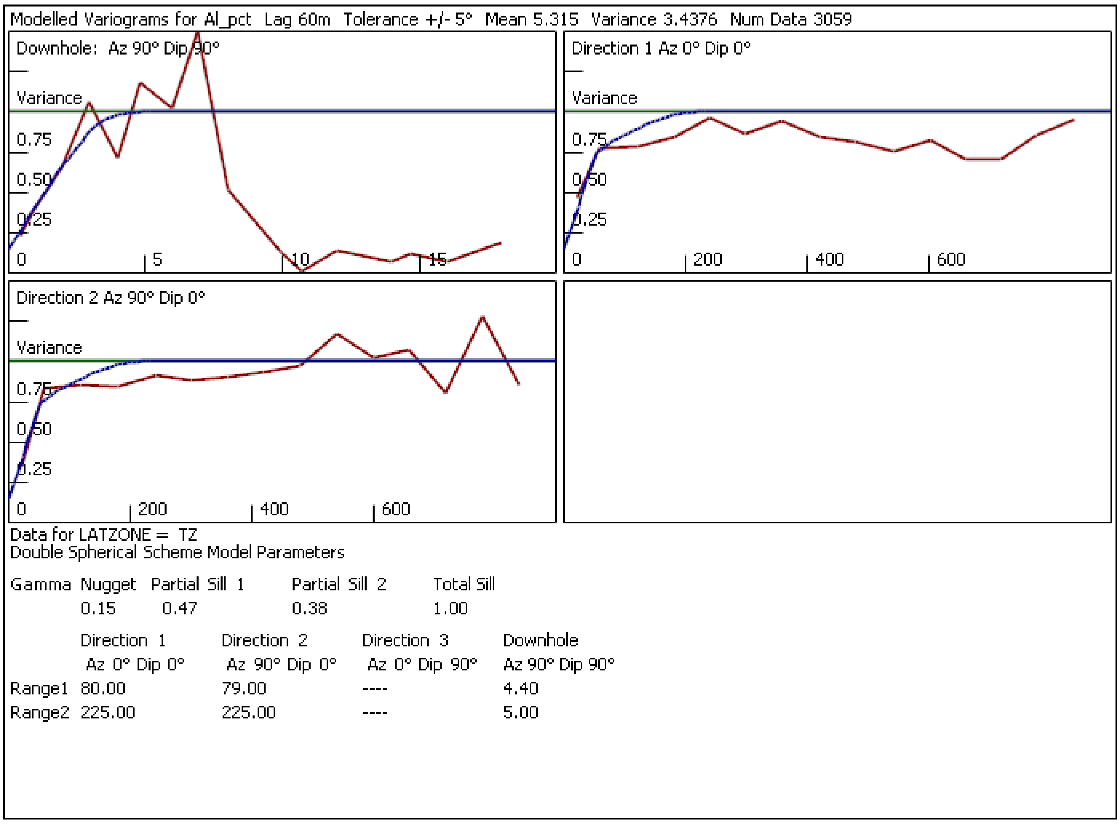
```
Modelled Variograms for Si_pct  Lag 60m  Tolerance +/- 5°  Mean 8.097  Variance 31.2057  Num Data 2111

Data for LATZONE =  TZ
Double Spherical Scheme Model Parameters

Gamma  Nugget   Partial Sill 1     Partial Sill 2     Total Sill
       0.24       0.06              0.70               1.00

       Direction 1      Direction 2      Direction 3      Downhole
        Az 0° Dip 0°     Az 90° Dip 0°    Az 0° Dip 90°   Az 90° Dip 90°
Range1  110.00          128.57           ----             5.30
Range2  290.00          267.00           ----             10.30
```

Figure 14-67: TZ Si



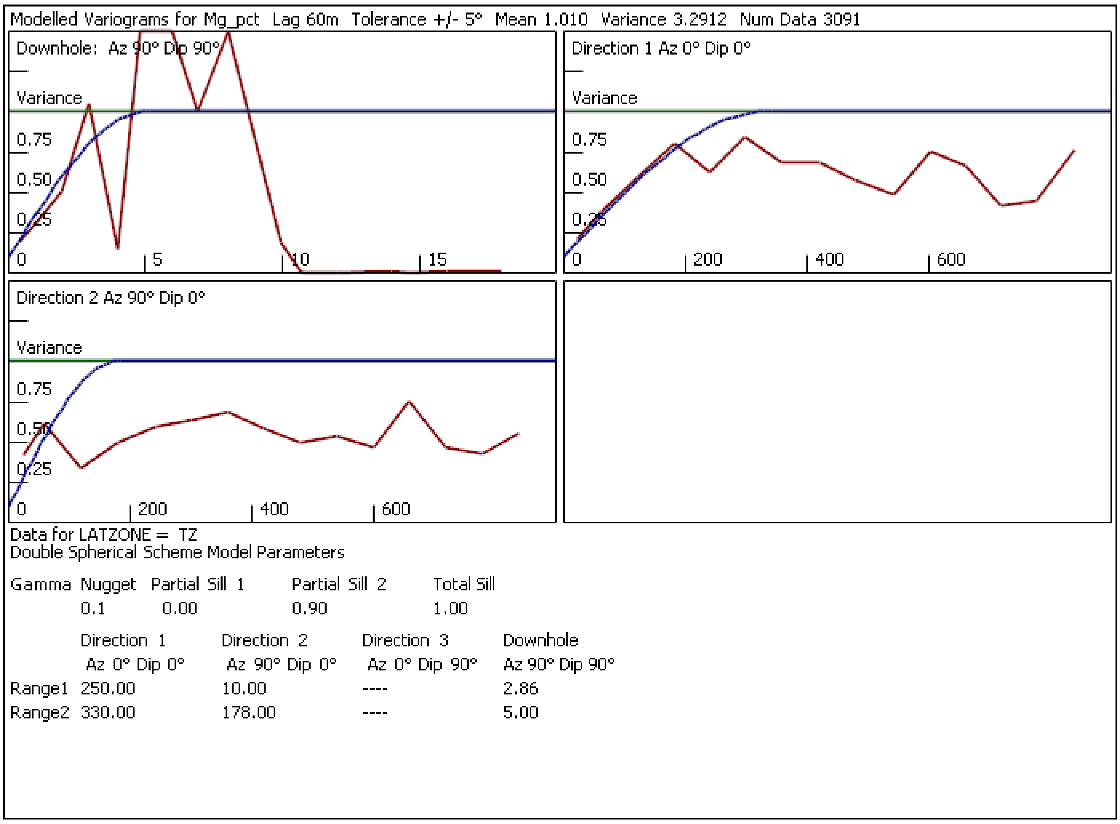
```
Modelled Variograms for Al_pct  Lag 60m  Tolerance +/- 5°  Mean 5.315  Variance 3.4376  Num Data 3059
```

```
Data for LATZONE = TZ
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2      Total Sill
       0.15      0.47              0.38                1.00

       Direction 1         Direction 2         Direction 3         Downhole
        Az 0° Dip 0°        Az 90° Dip 0°       Az 0° Dip 90°      Az 90° Dip 90°
Range1  80.00              79.00               ----                4.40
Range2  225.00             225.00              ----                5.00
```

Figure 14-68: TZ Al



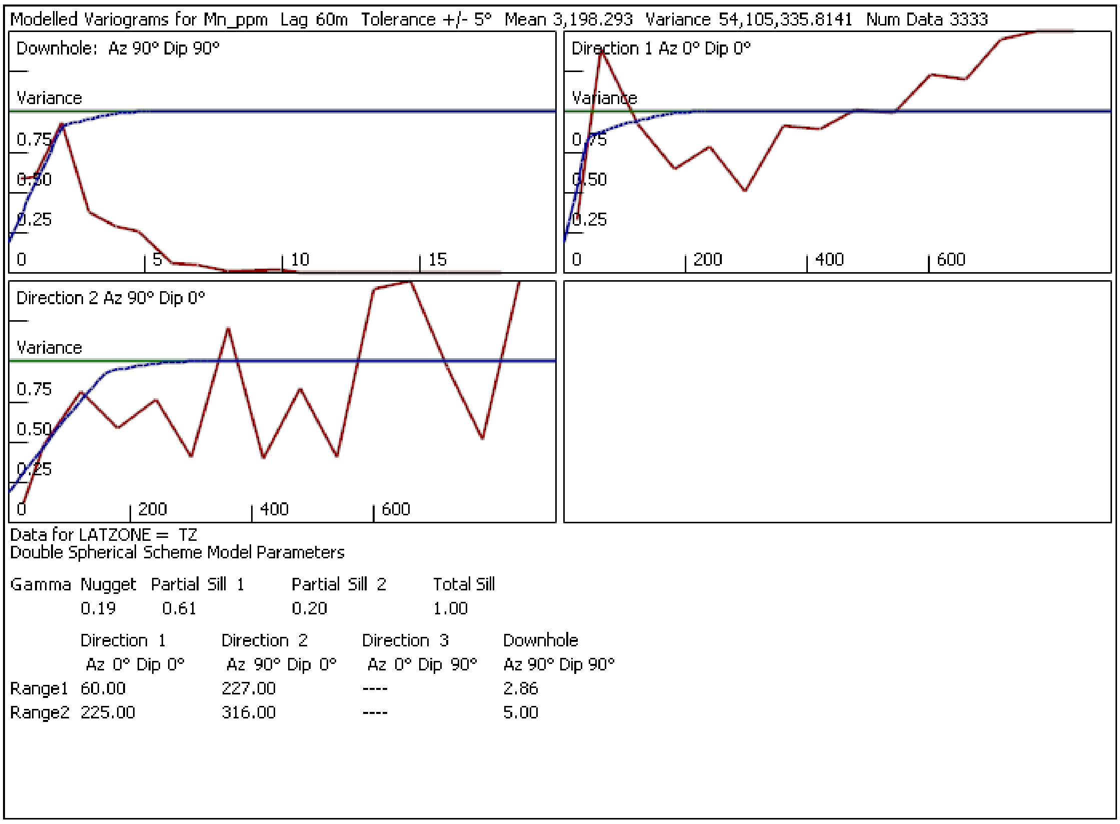
```
Modelled Variograms for Mg_pct  Lag 60m  Tolerance +/- 5°  Mean 1.010  Variance 3.2912  Num Data 3091
```

```
Data for LATZONE = TZ
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2      Total Sill
       0.1       0.00              0.90                1.00

       Direction 1         Direction 2         Direction 3         Downhole
        Az 0° Dip 0°        Az 90° Dip 0°       Az 0° Dip 90°      Az 90° Dip 90°
Range1  250.00             10.00               ----                2.86
Range2  330.00             178.00              ----                5.00
```

Figure 14-69: TZ Mg

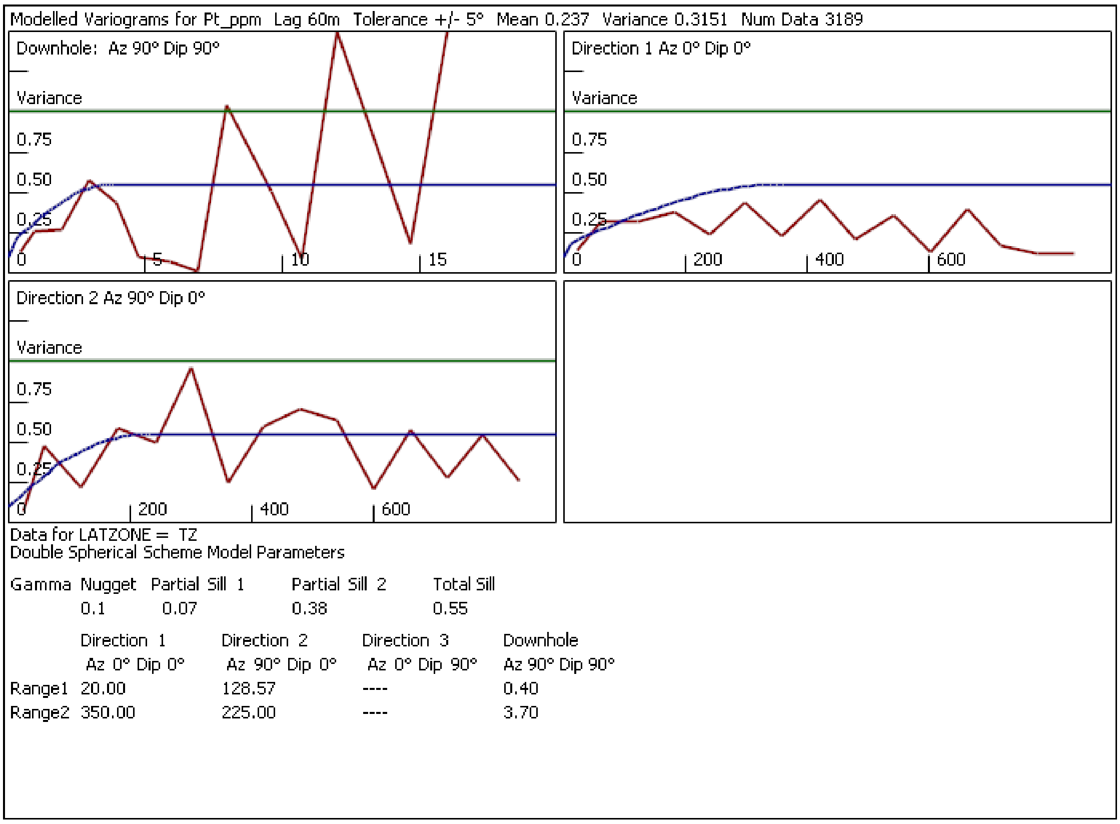


Data for LATZONE = TZ
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.19	0.61	0.20	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	60.00	227.00	----	2.86
Range2	225.00	316.00	----	5.00

Figure 14-70: TZ Mn

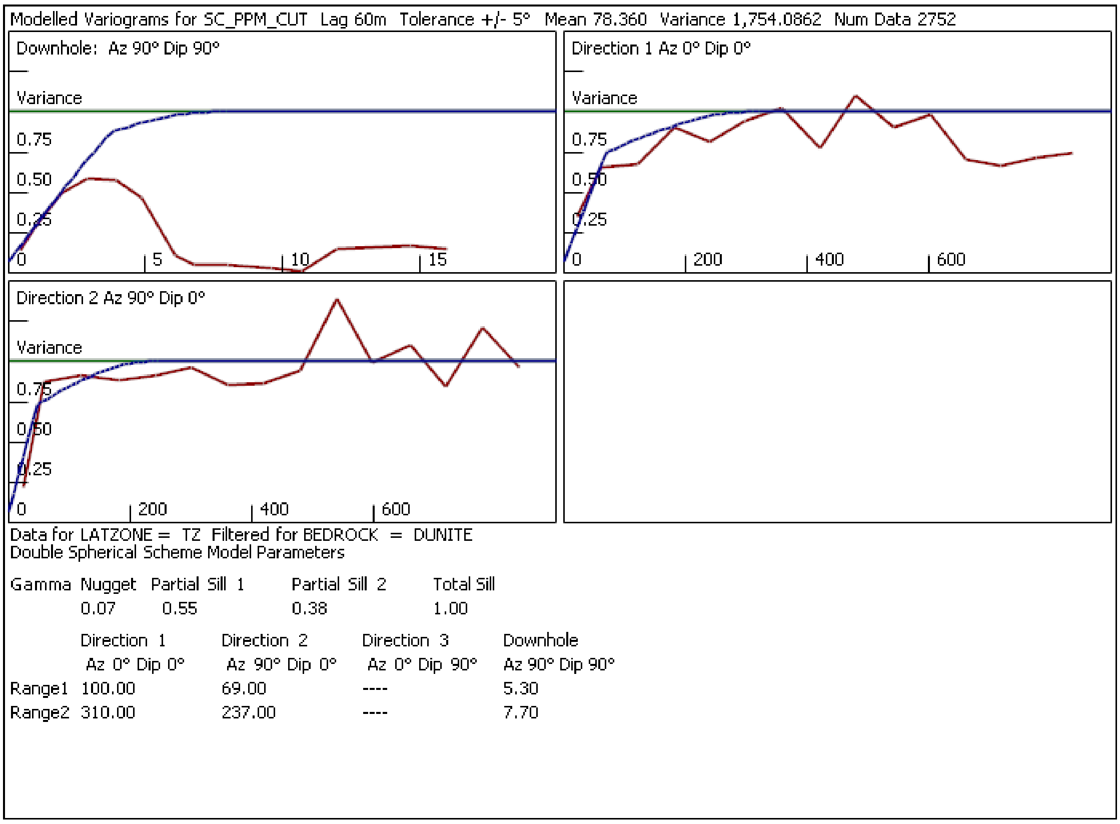


Data for LATZONE = TZ
Double Spherical Scheme Model Parameters

Gamma	Nugget	Partial Sill 1	Partial Sill 2	Total Sill
	0.1	0.07	0.38	0.55

	Direction 1 Az 0° Dip 0°	Direction 2 Az 90° Dip 0°	Direction 3 Az 0° Dip 90°	Downhole Az 90° Dip 90°
Range1	20.00	128.57	----	0.40
Range2	350.00	225.00	----	3.70

Figure 14-71: TZ Pt



```
Data for LATZONE =  TZ  Filtered for BEDROCK  =  DUNITE
Double Spherical Scheme Model Parameters

Gamma  Nugget  Partial Sill 1      Partial Sill 2      Total Sill
       0.07      0.55              0.38                1.00

       Direction 1         Direction 2         Direction 3         Downhole
        Az 0° Dip 0°        Az 90° Dip 0°       Az 0° Dip 90°      Az 90° Dip 90°
Range1  100.00             69.00               ----                5.30
Range2  310.00             237.00              ----                7.70
```

Figure 14-72: TZ Sc

14.1.8 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 (see Figure 14-73).

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 scandium, platinum, iron, silicon, magnesium, aluminium and manganese were generated using Ordinary Kriging. Copper, zinc, chromium, arsenic, gold, palladium and calcium were estimated using an anisotropic inverse distance squared interpolation method.

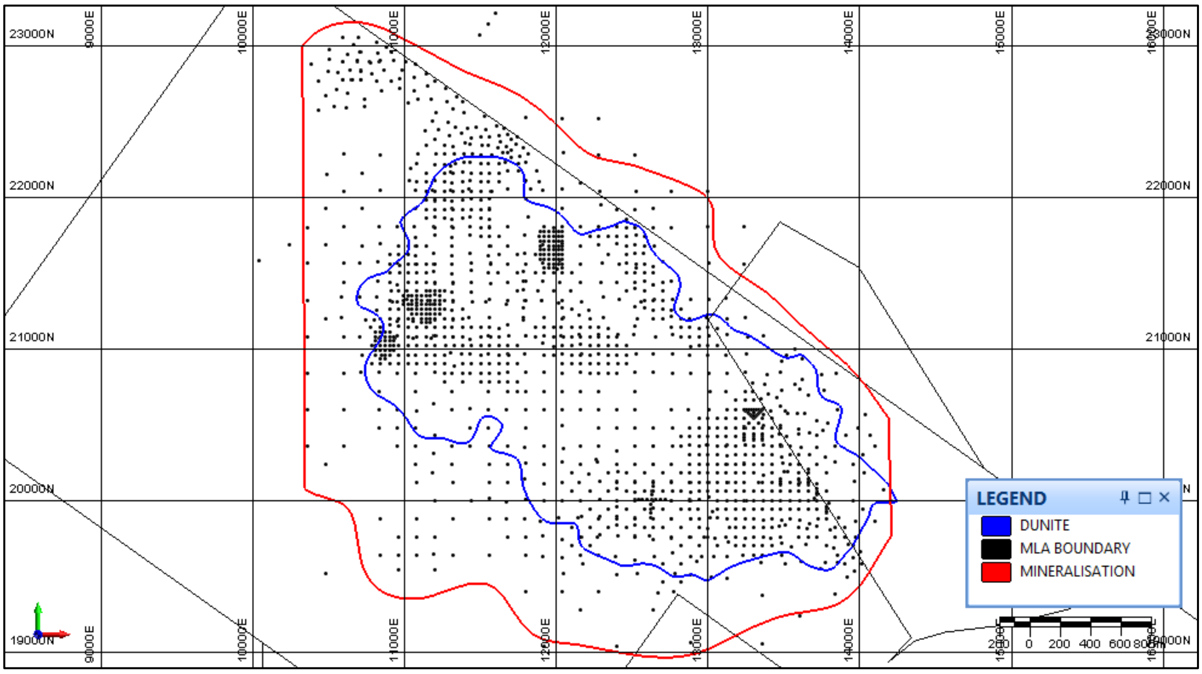


Figure 14-73: Drill Hole Locations, Dunite Limit and Mineralisation Limits

14.1.9 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 selective mining unit (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 Clean TeQ Sunrise, spacing varies from small areas of very close spacing (<15 m) to wide spaced (>200 m) areas. There are clear areas with approximate grids of 60x60 m and 120x120 m, 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 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 sub-cells. There is also sub-celling to follow topographic and domain boundaries, so this is the normal case in rock model setup.

For the Clean TeQ Sunrise 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 (see Figure 14-74).

An additional reason for minimising block size came out of a comparison of previous kriged models with a conditional simulation 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 (10 m for 60x60 and 20 m for 120x120). Whilst blocks of this size may make the block grade distribution mover closer to the conditional simulation 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 2 m was based on advice about a likely open pit working bench height.

The figures 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 (Figure 14-75 to Figure 14-79).

Following this, areas were digitised to control the size of blocks in the rock model. The grids are shown on the following pages.

Table 14-9 below shows the relative proportions of each block size; interestingly the main mineralised zones (GZ, and to a lesser extent TZ and SGZ) are better drilled and have higher representation of the smaller block sizes.

Table 14-9: 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%

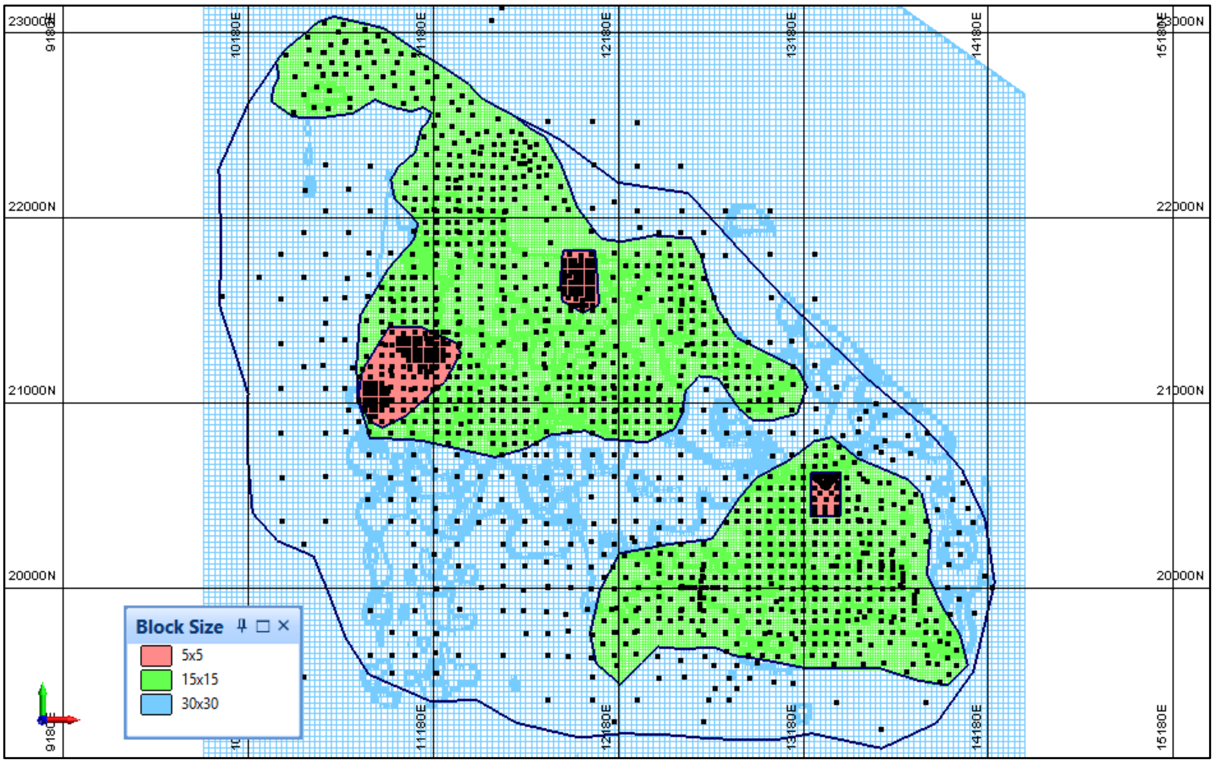


Figure 14-74: Final Block Size Distribution

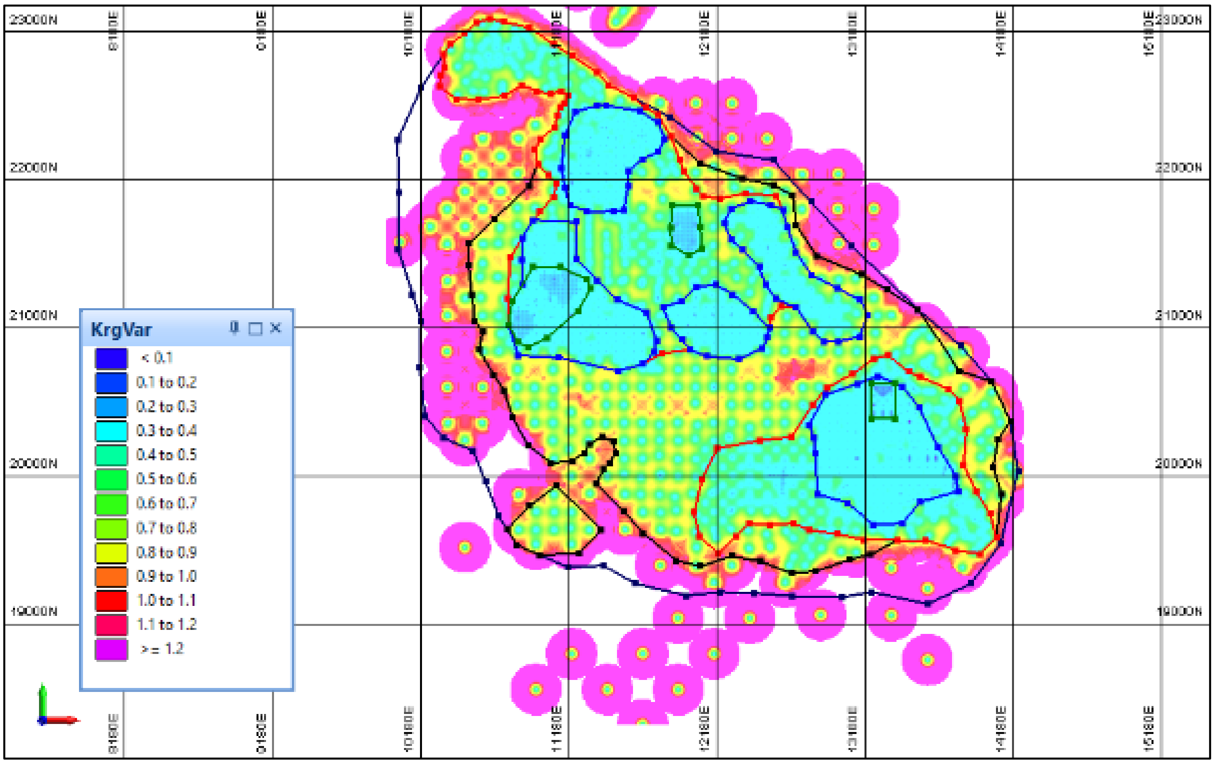


Figure 14-75: 2D “kriging variance” of collar locations

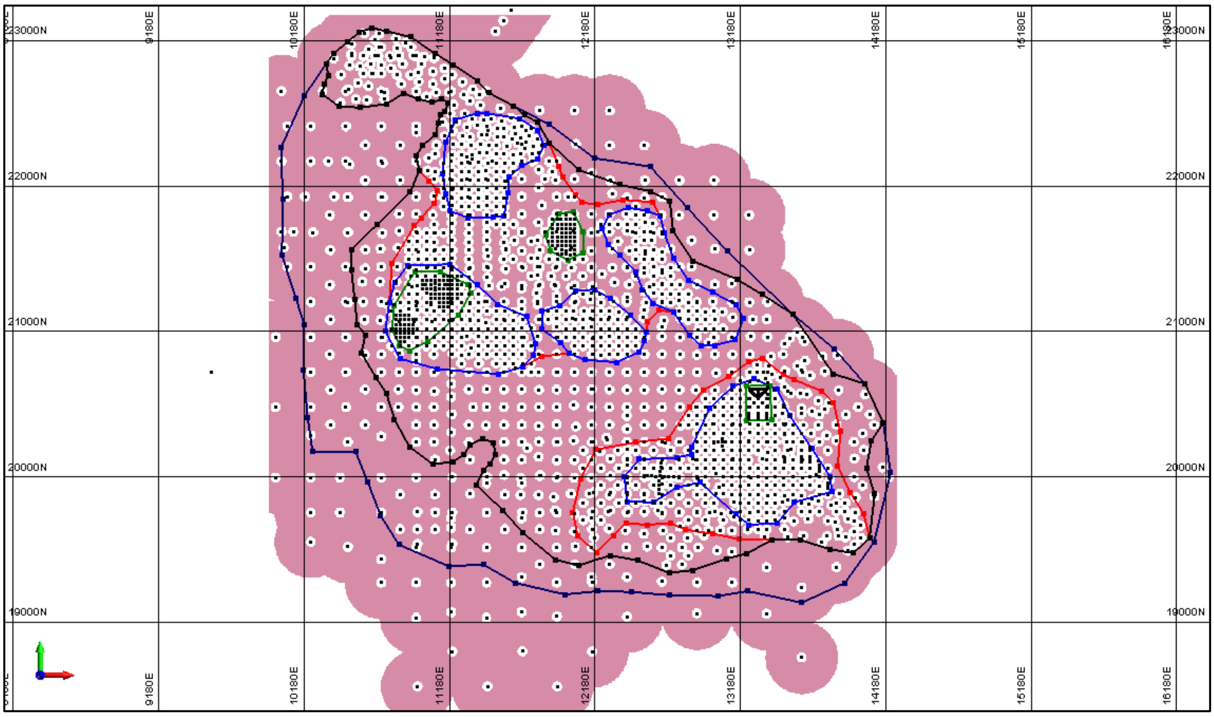


Figure 14-76: Average Distance < 35m (white areas)

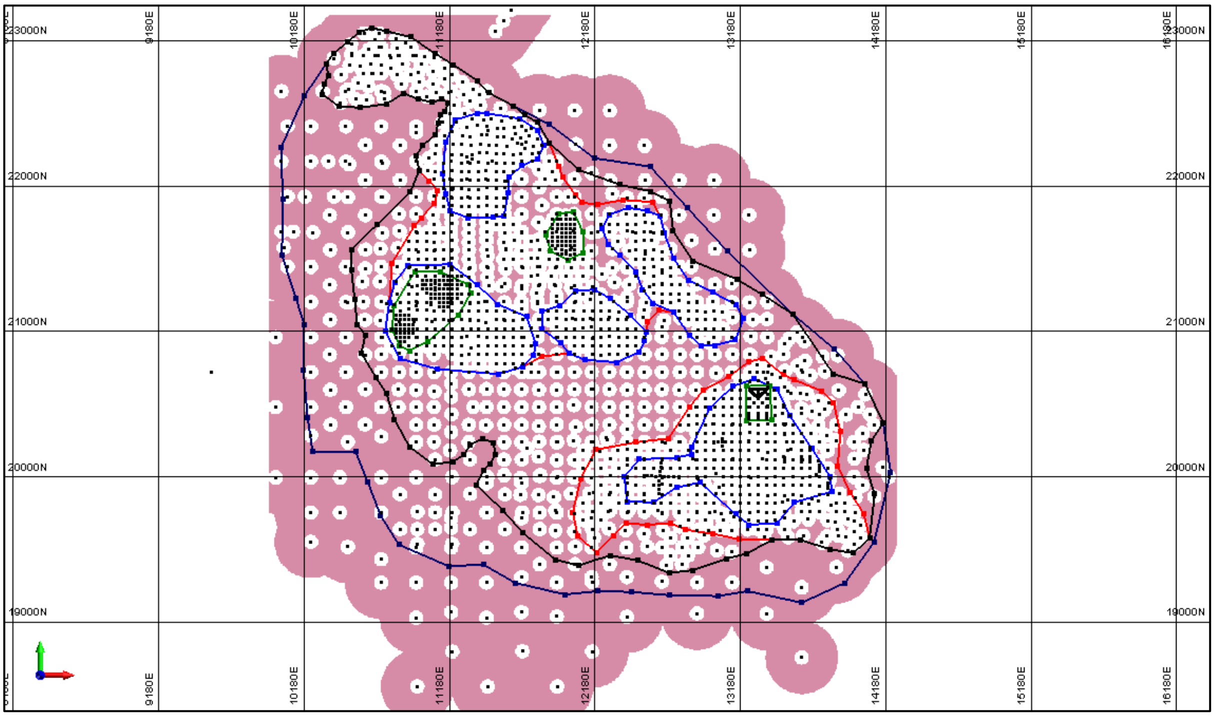


Figure 14-77: Average Distance < 50m (white areas)

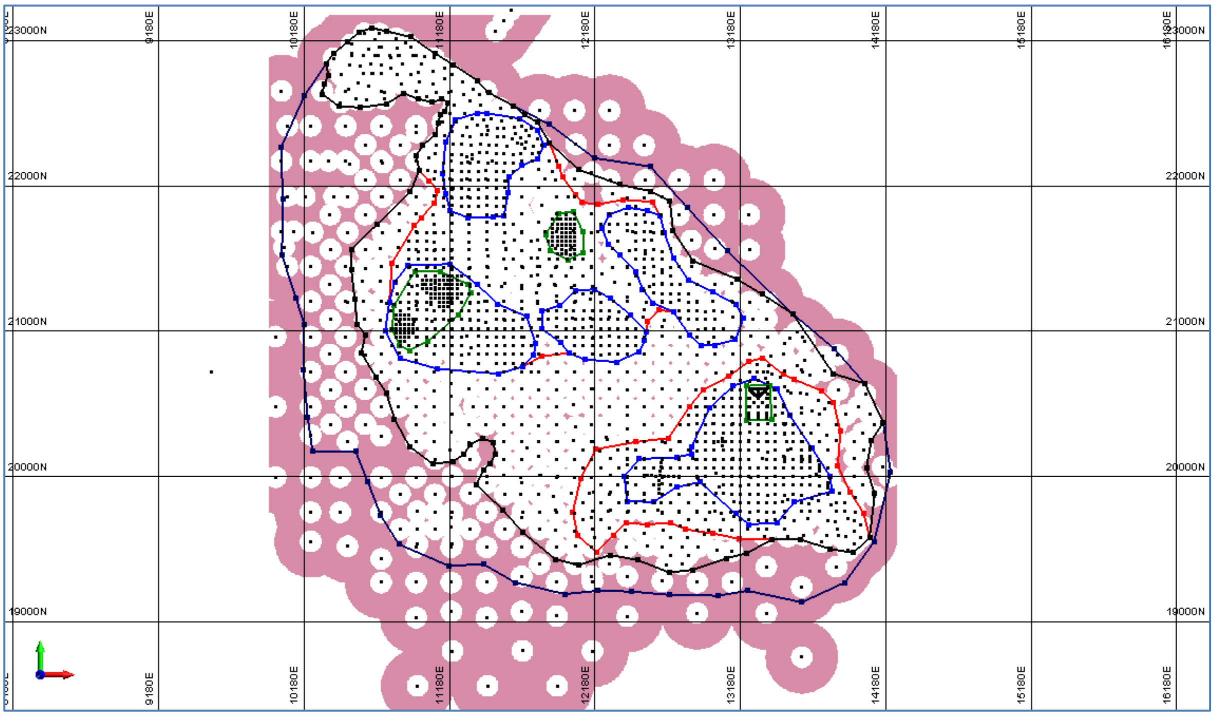


Figure 14-78: Average Distance < 75m (white areas)

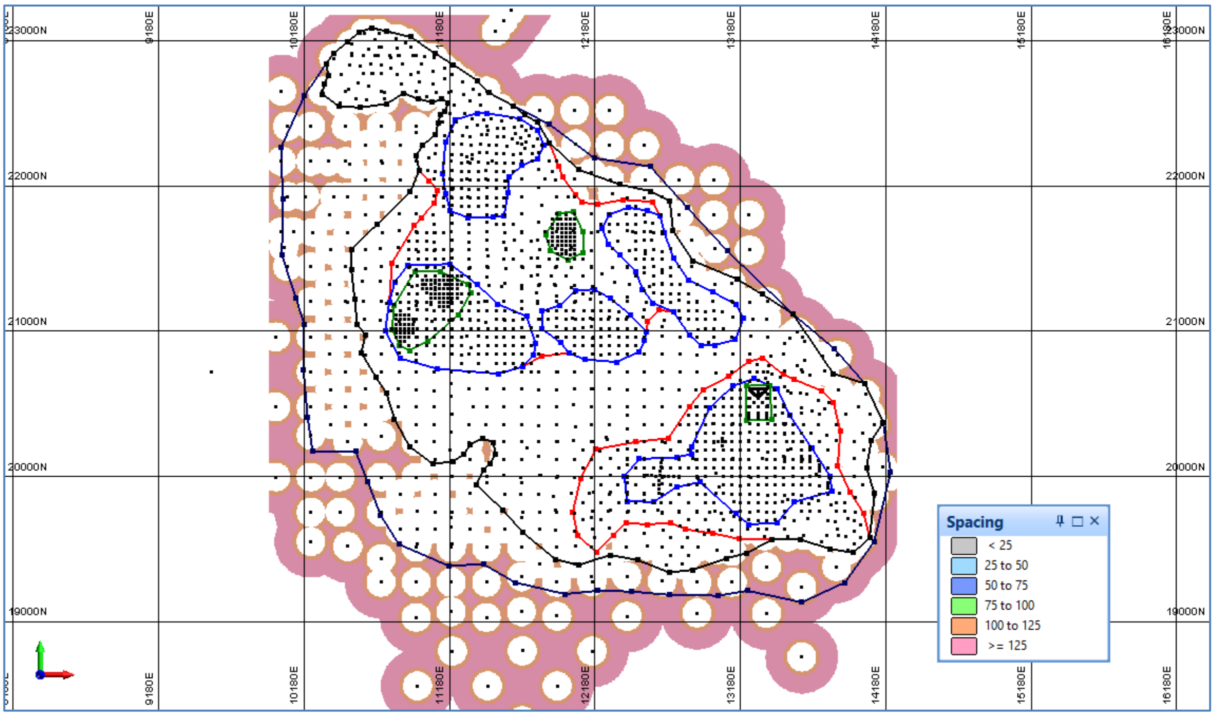


Figure 14-79: Average Distance < 100m (white areas)

14.1.10 Rock Model

Laterite zone boundaries have been re-interpreted as strings on north–south section lines based on 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 silicified goethite zone (SGZ)

The wireframe surfaces have then been smoothed by the IM 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, to prevent extrapolation below the base of drill holes in saprolite, a Base of SAP surface was generated parallel to the Base of SGZ surface and 8 m 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 three-dimensional rock model. This format is shown in Table 14-10 and Table 14-11.

Table 14-10: 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
9,955	14,395	18,315	23,295	221	329	30	30	2	149	167	55	0.5	0.5	0.2

Table 14-11: Block Model Origin and Block Size Parameters - 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
9,940	14,410	18,300	23,310	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 in Figure 14-80 to Figure 14-82.

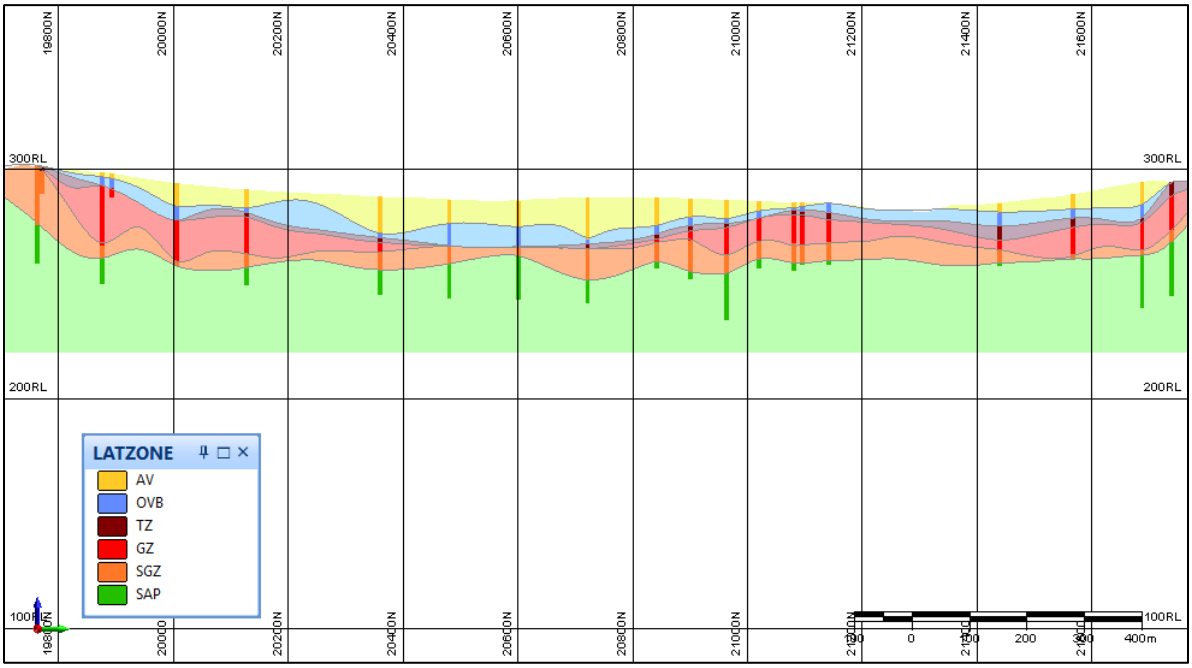


Figure 14-80: Typical Easting Section

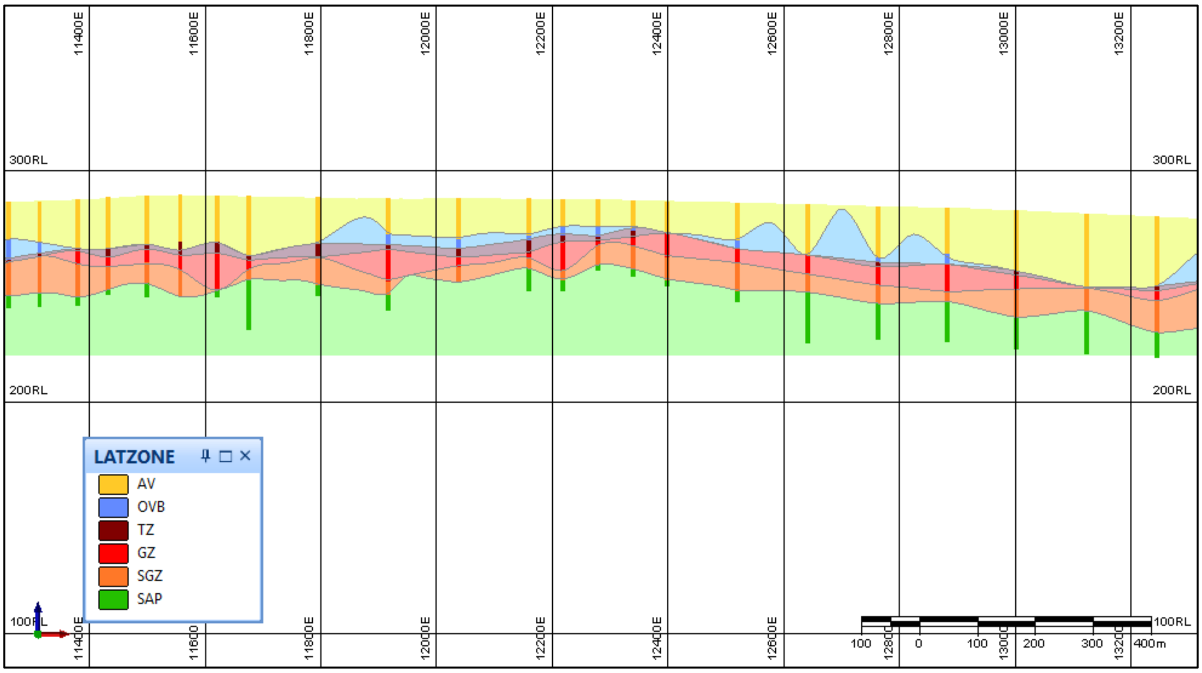


Figure 14-81: Typical Northing Section

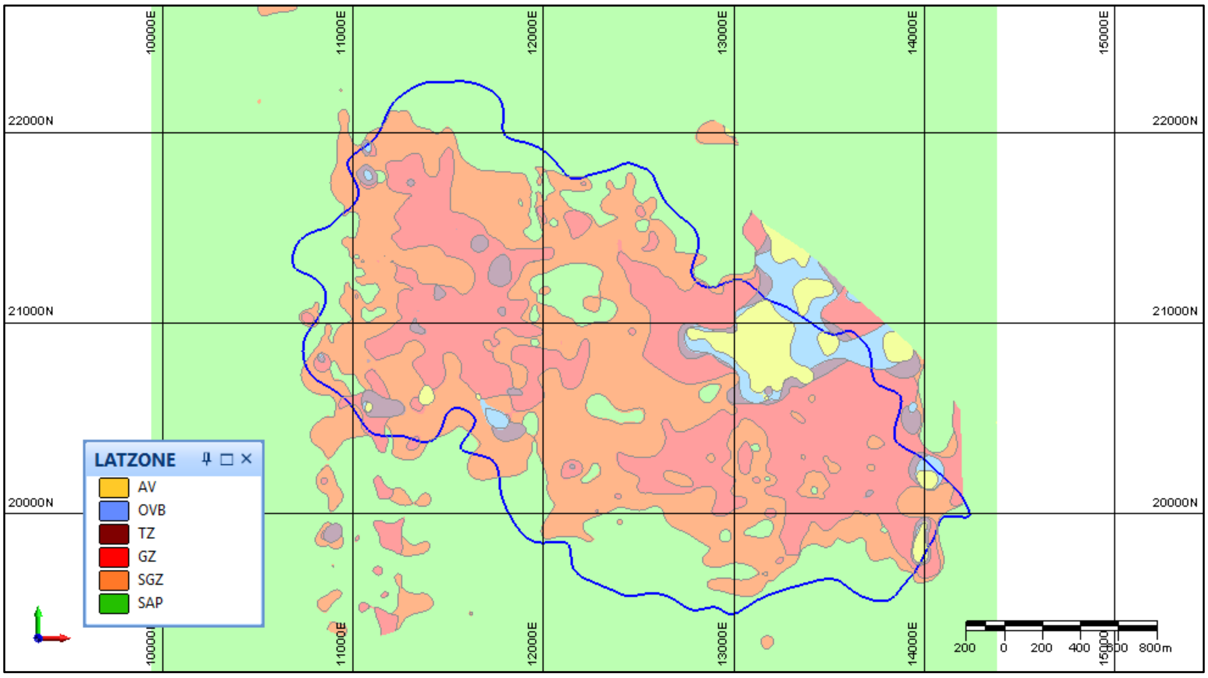


Figure 14-82: Typical Level Plan

14.1.11 Interpolation Parameters

Search parameters are based on the variogram ranges, with the first search ellipse pass approximating the 60x60 drill spacing (and within the first range of the double spherical scheme variogram models) and the second pass designed to populate blocks from data on the 120x120 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 Figure 14-83.

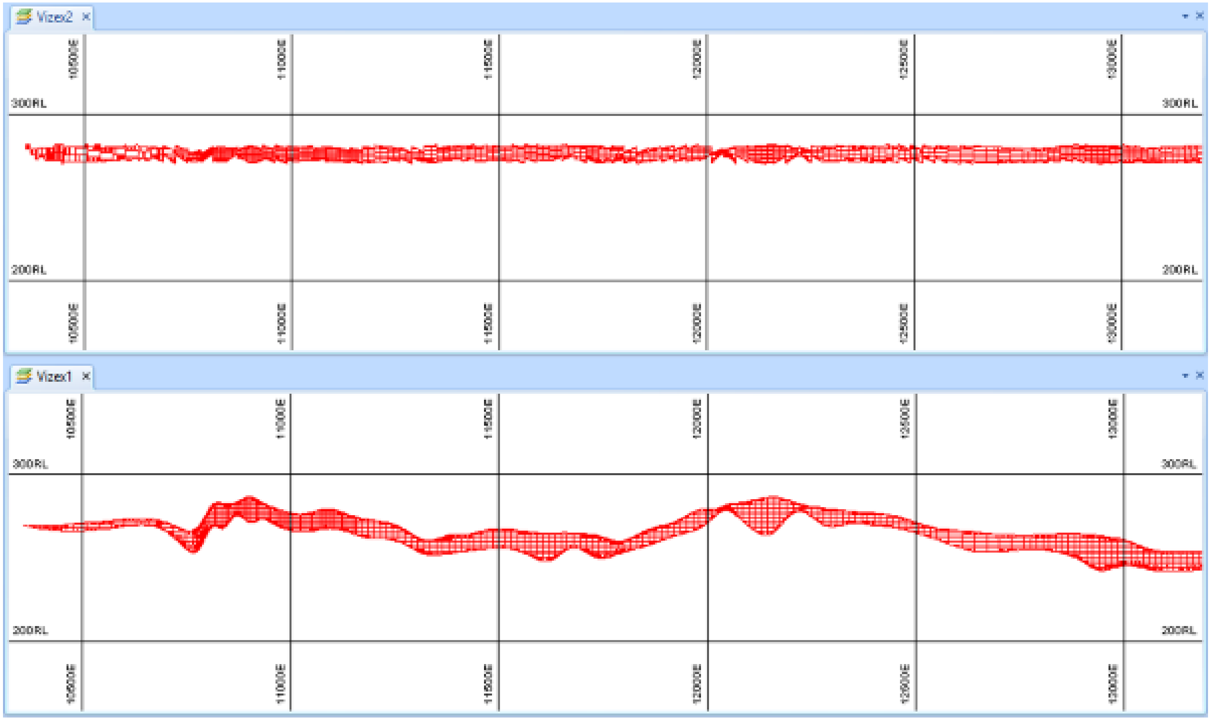


Figure 14-83: Typical Section showing unfolded (top) vs in-situ blocks

Search parameters are the same for Ordinary Kriging and inverse distance interpolation are shown in Table 14-12.

Table 14-12: Search Parameters

Pass	Blocks Size	Search Ellipse			Samples		Holes	Per Hole	
		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

14.1.12 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 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 cobalt GZ domain.

Review of the spatial location of the high-grade cobalt 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 cobalt shells are illustrated in Figure 14-84 and Figure 14-85.

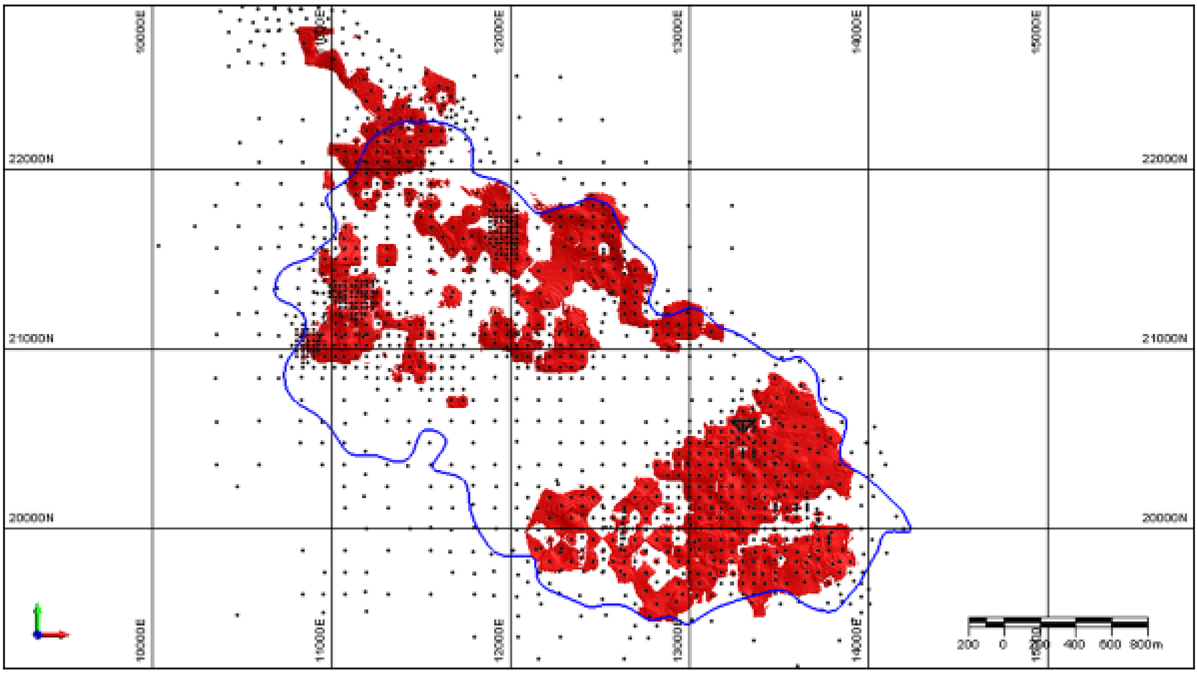


Figure 14-84: High Grade Cobalt Plan View

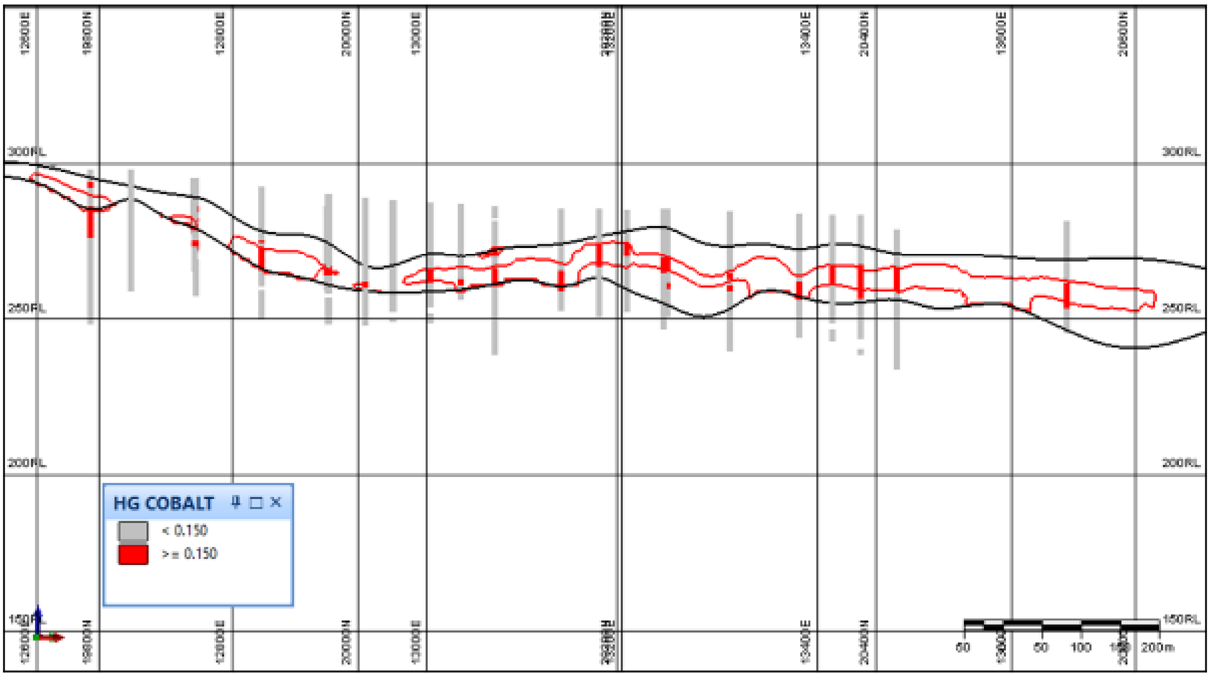


Figure 14-85: Typical Section of High Grade Cobalt Zone

14.1.13 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 cobalt has been taken but restricted to areas outside the dunite complex footprint. A threshold of 160 ppm scandium is used to define high-grade scandium and examples of the resultant domains are shown in Figure 14-86 and Figure 14-87.

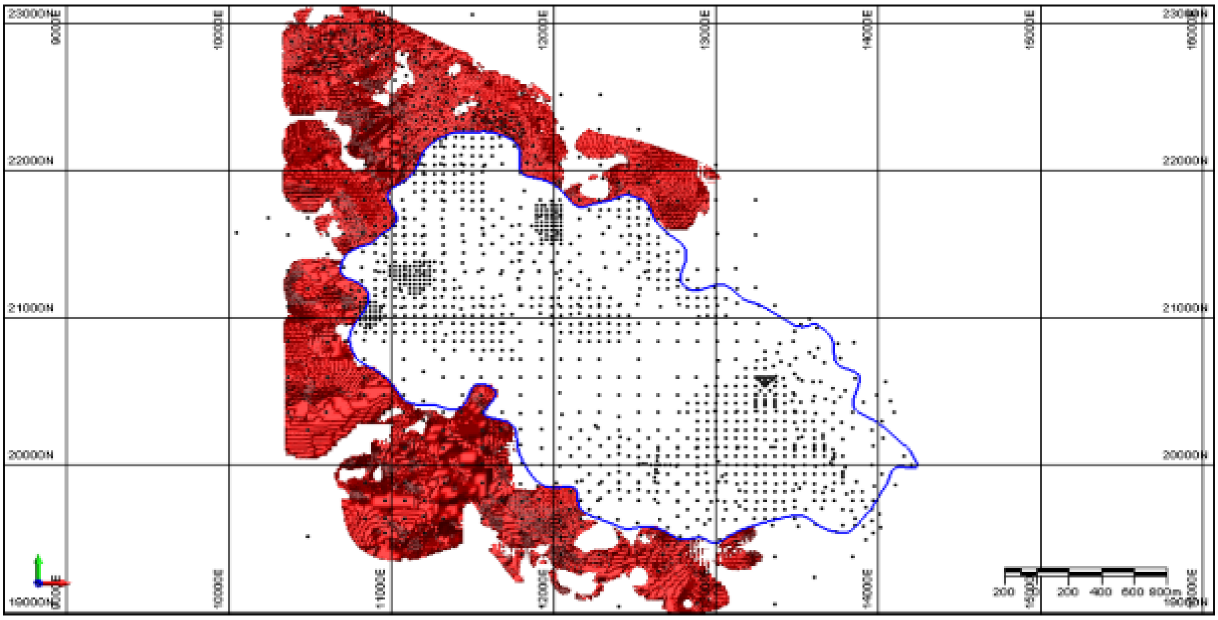


Figure 14-86: High Grade Scandium Domains

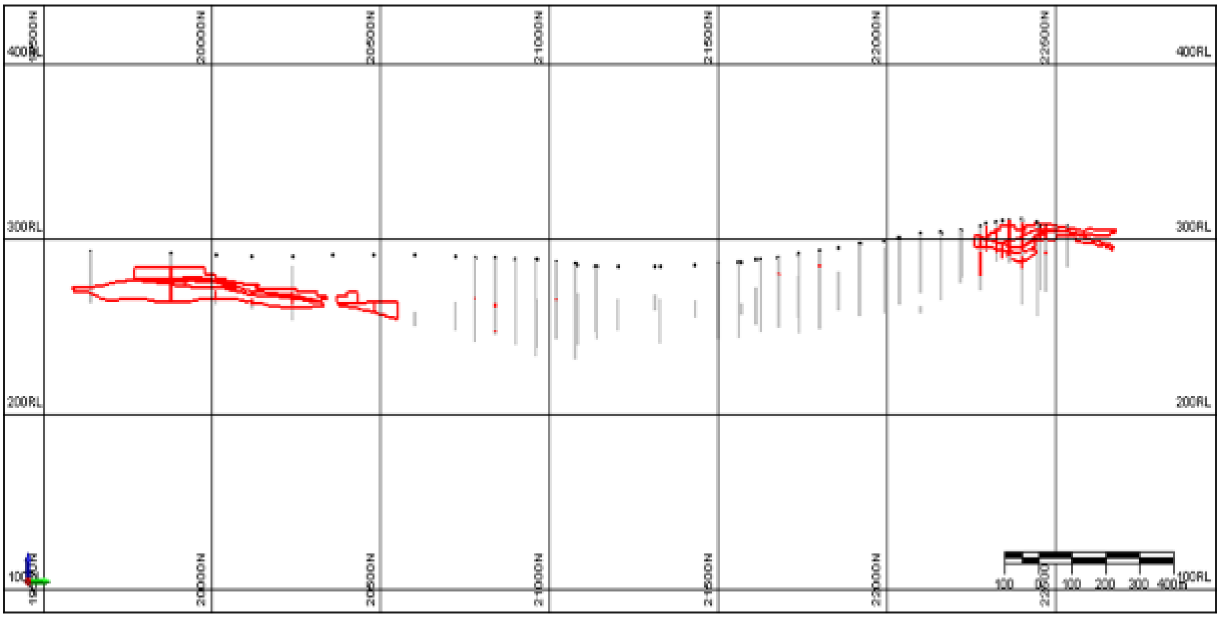


Figure 14-87: Typical Section of High Grade Scandium Zones

14.1.14 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 100m of drilling in mineralised zones by six 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 five 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 shown in Table 14-13.

Table 14-13: Density Assigned in Model

Domain	Code	Density
Alluvium	AV	1.80
Overburden	OVB	1.80
Transition zone	TZ	1.70
Goethitic zone	GZ	1.20
Silicified goethitic zone	SGZ	1.25
Saprolite	SAP	2.00

14.2 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 3D review.

All methods of validation are acceptable.

Swathe plots for cobalt and nickel are illustrated in Figure 14-88 to Figure 14-95.

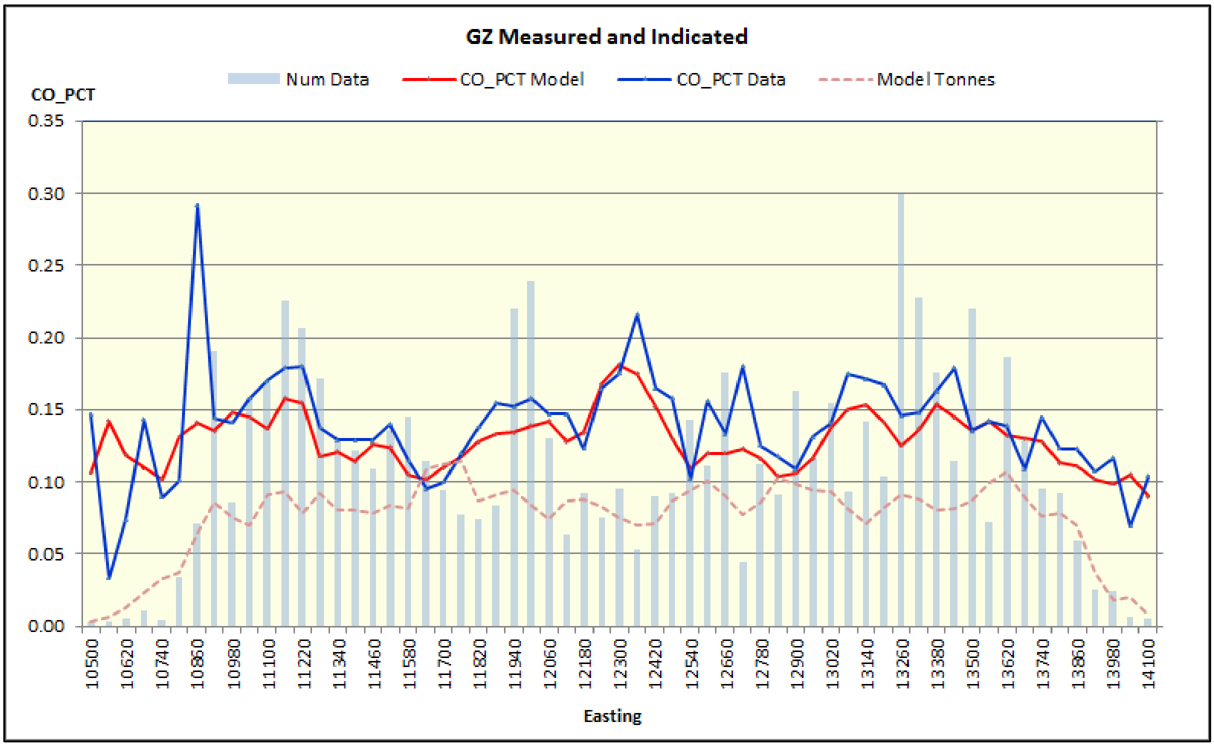


Figure 14-88: GZ Co by Easting

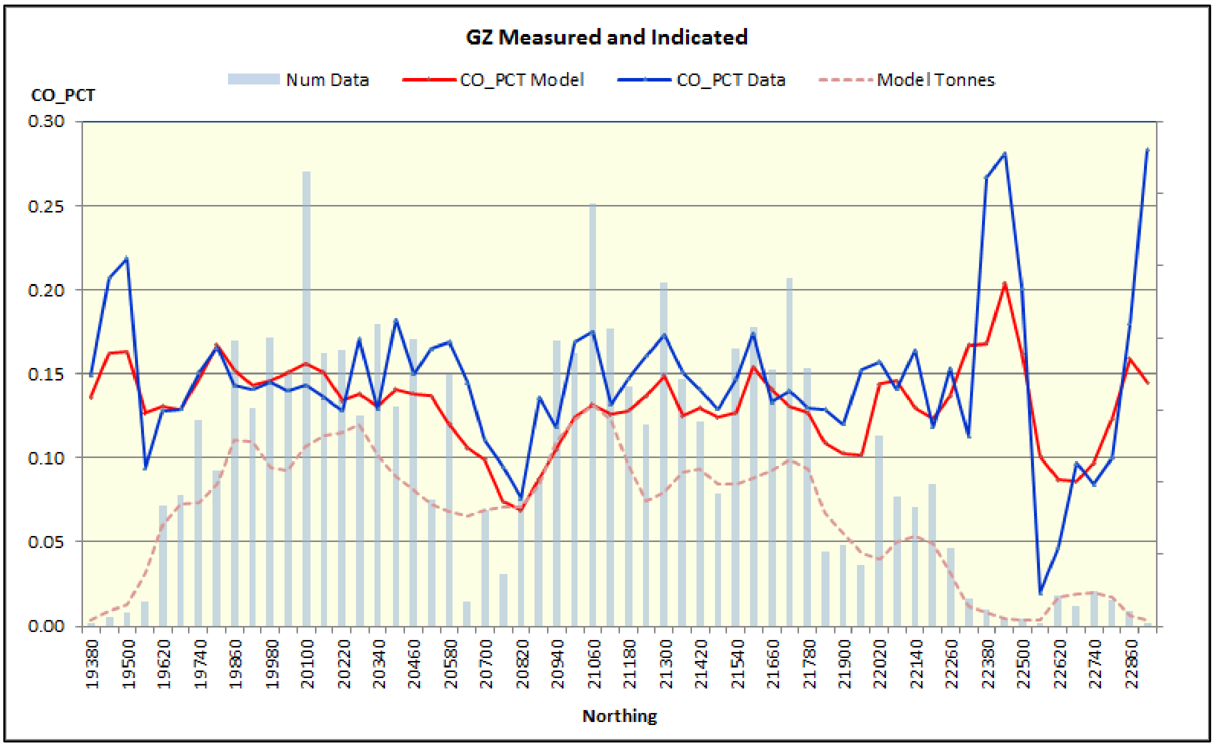


Figure 14-89: GZ Co by Northing

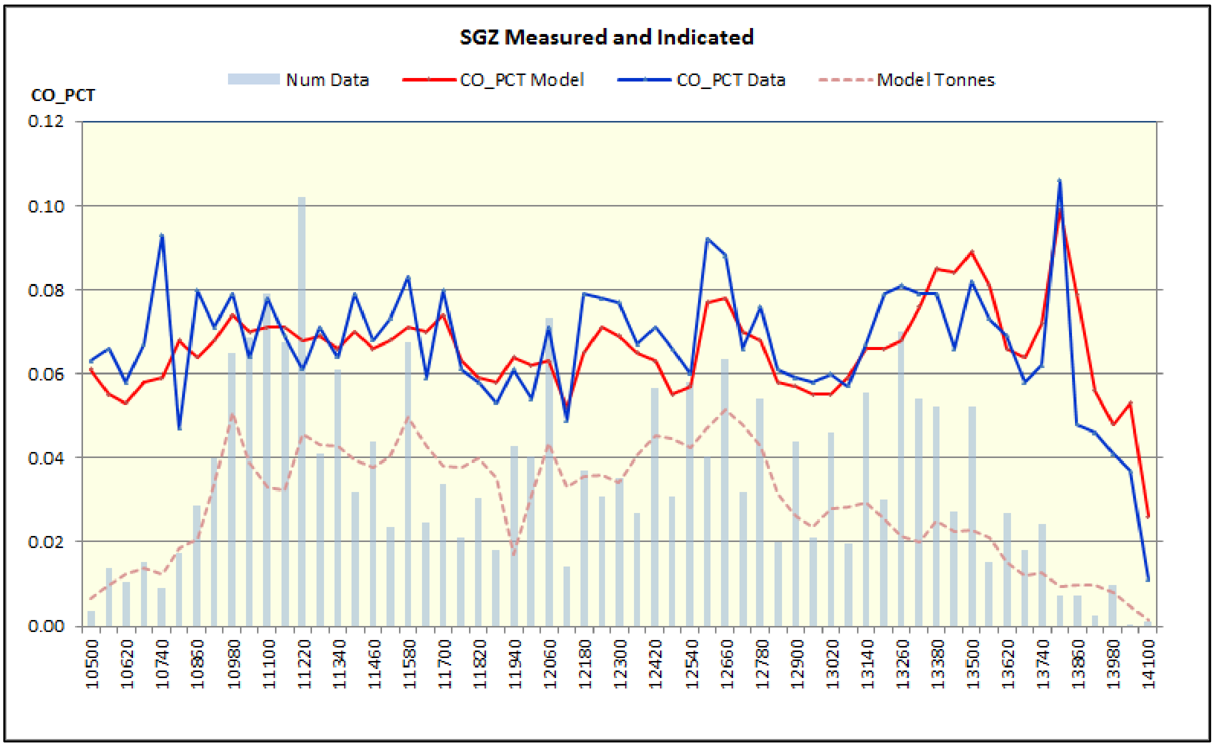


Figure 14-90: SGZ Co by Easting

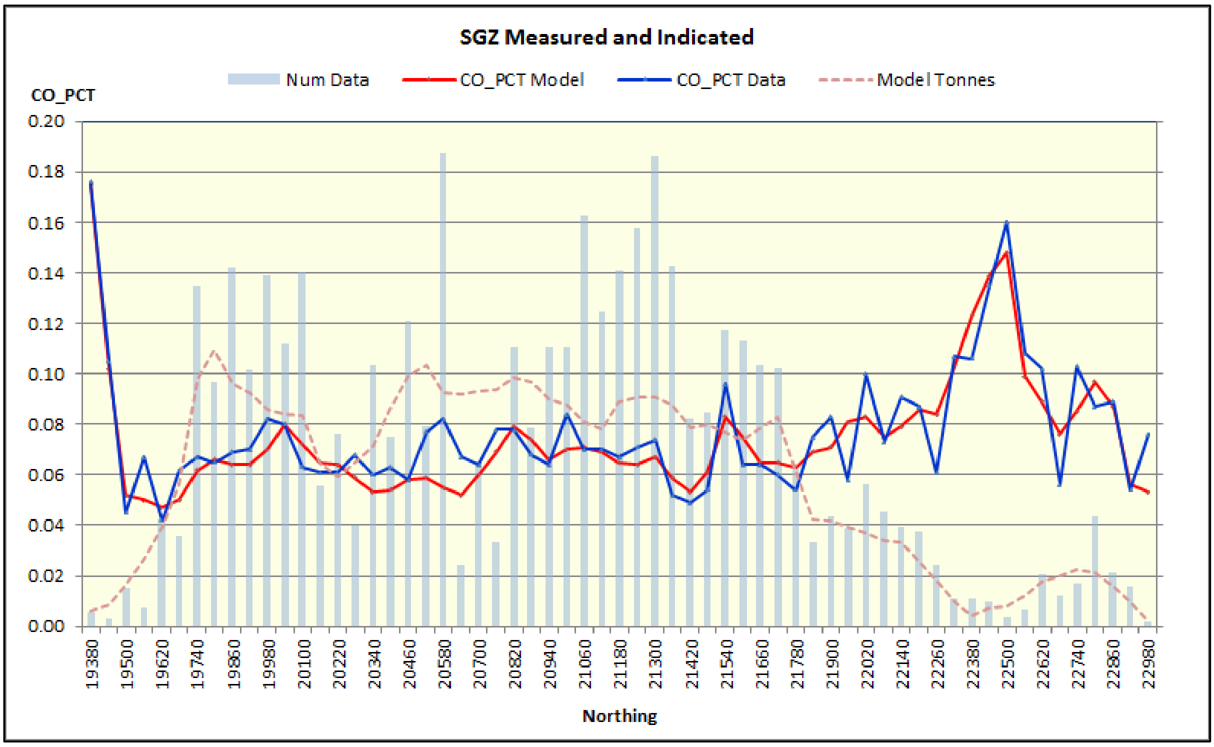


Figure 14-91: SGZ Co by Northing

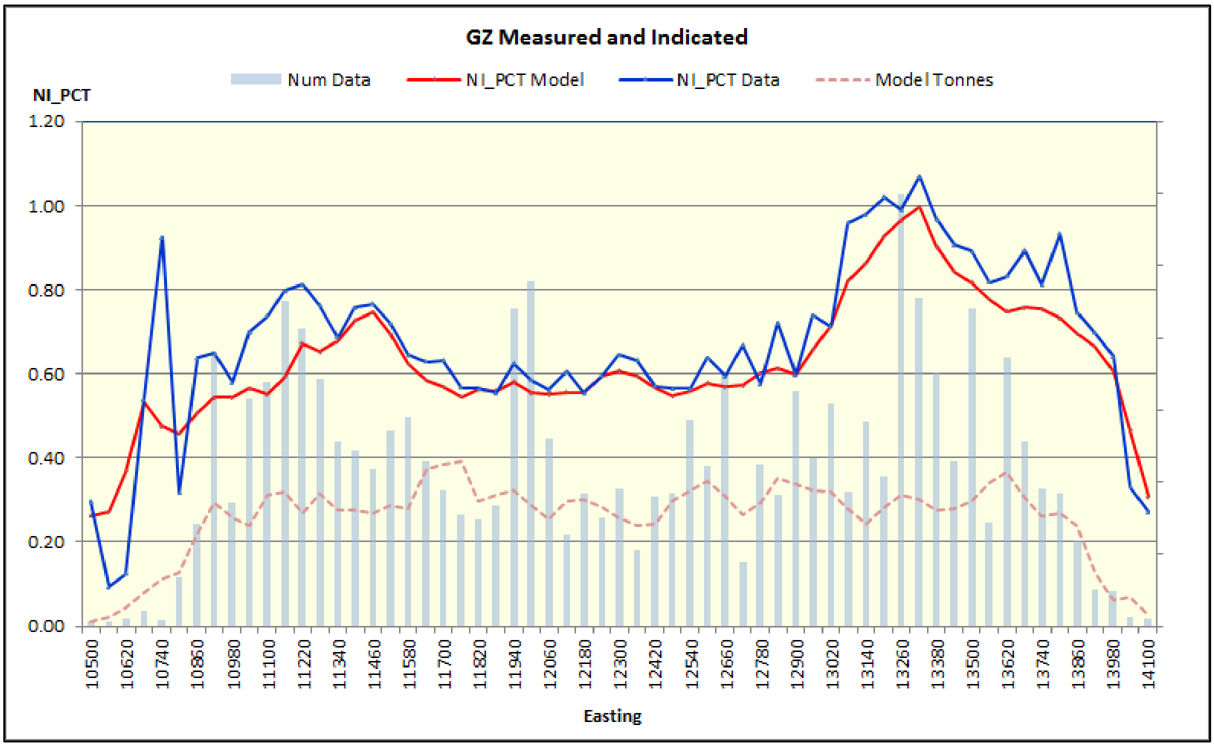


Figure 14-92: GZ Ni by Easting

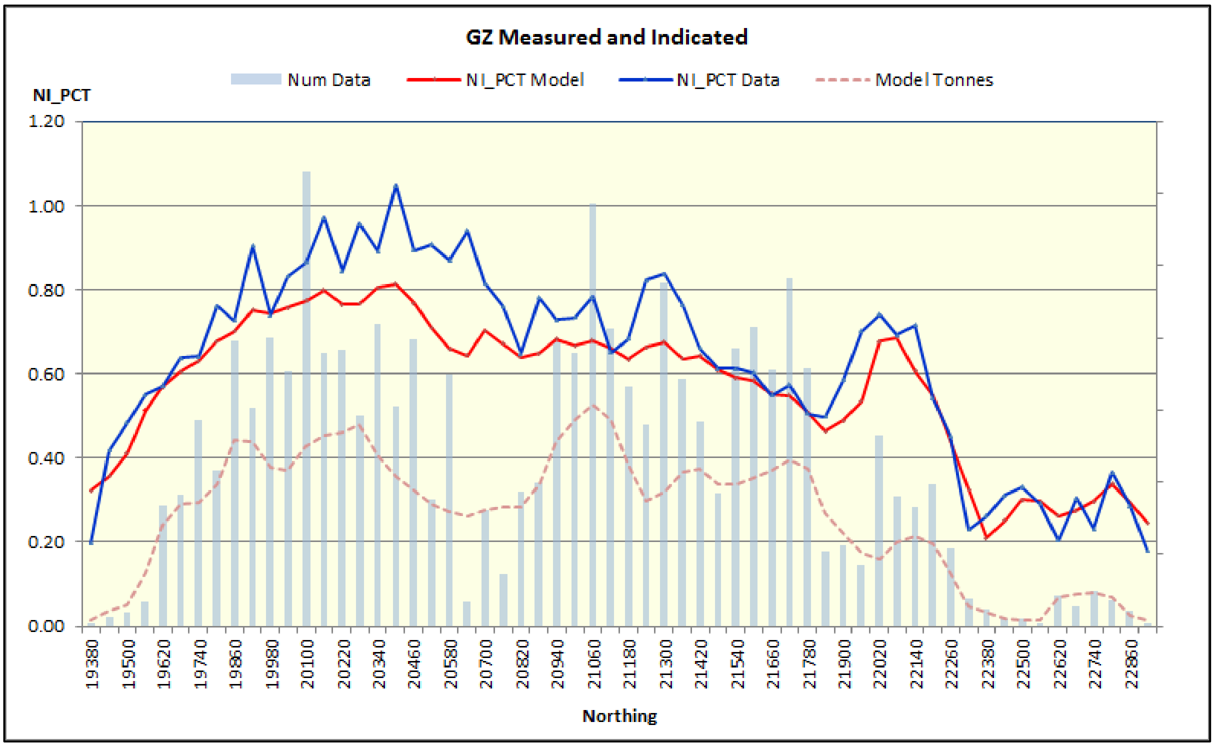


Figure 14-93: GZ Ni by Easting

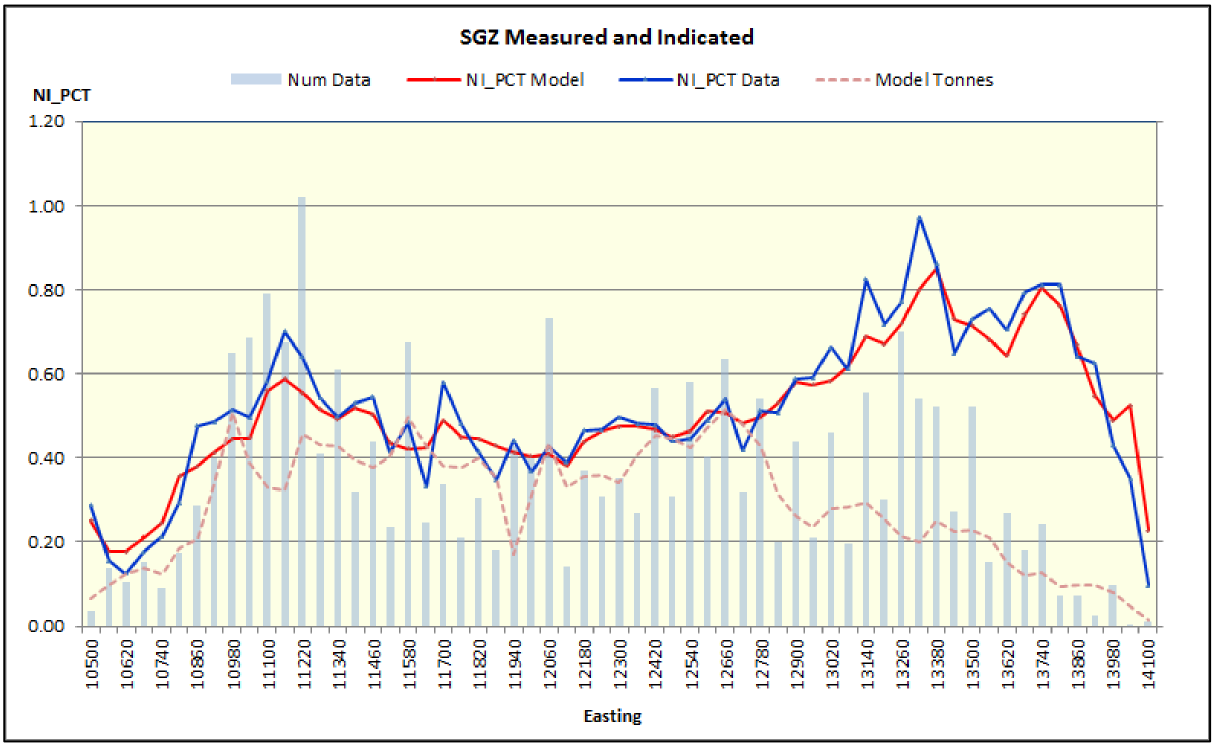


Figure 14-94: SGZ Ni by Easting

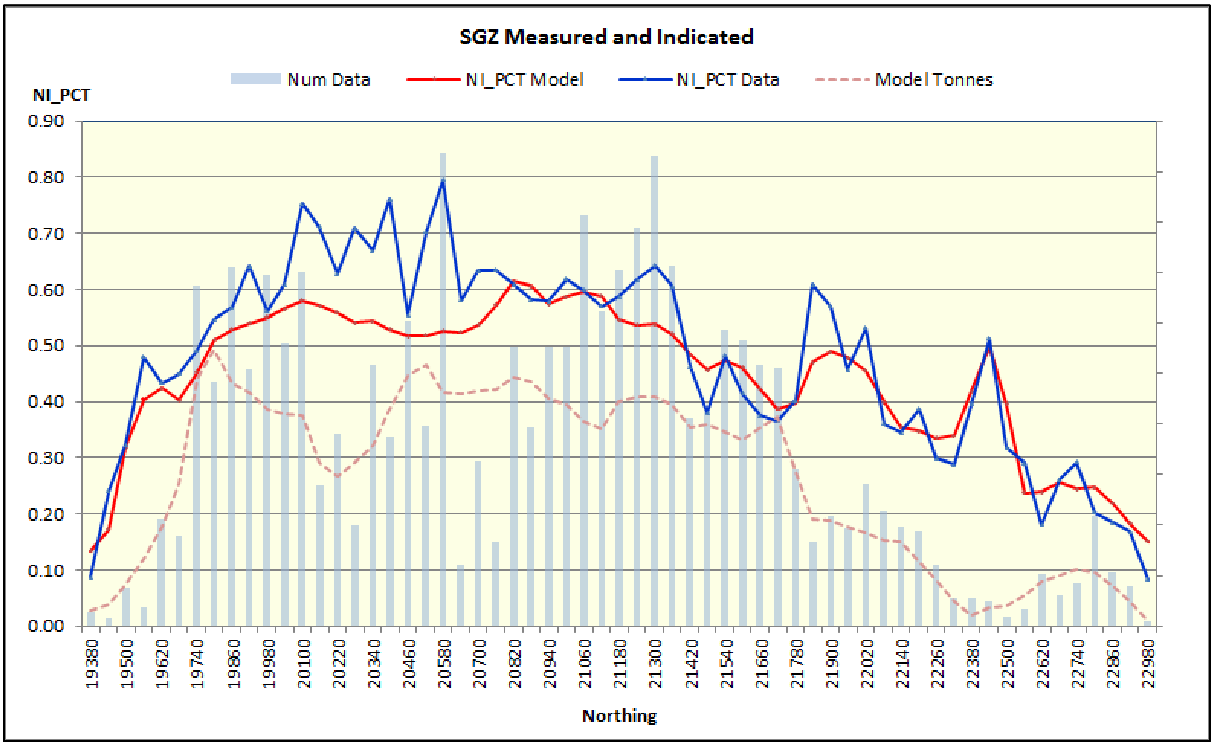


Figure 14-95: SGZ Ni by Northing

Declustered data vs model correlation plots are shown in Figure 14-96 to Figure 14-99.

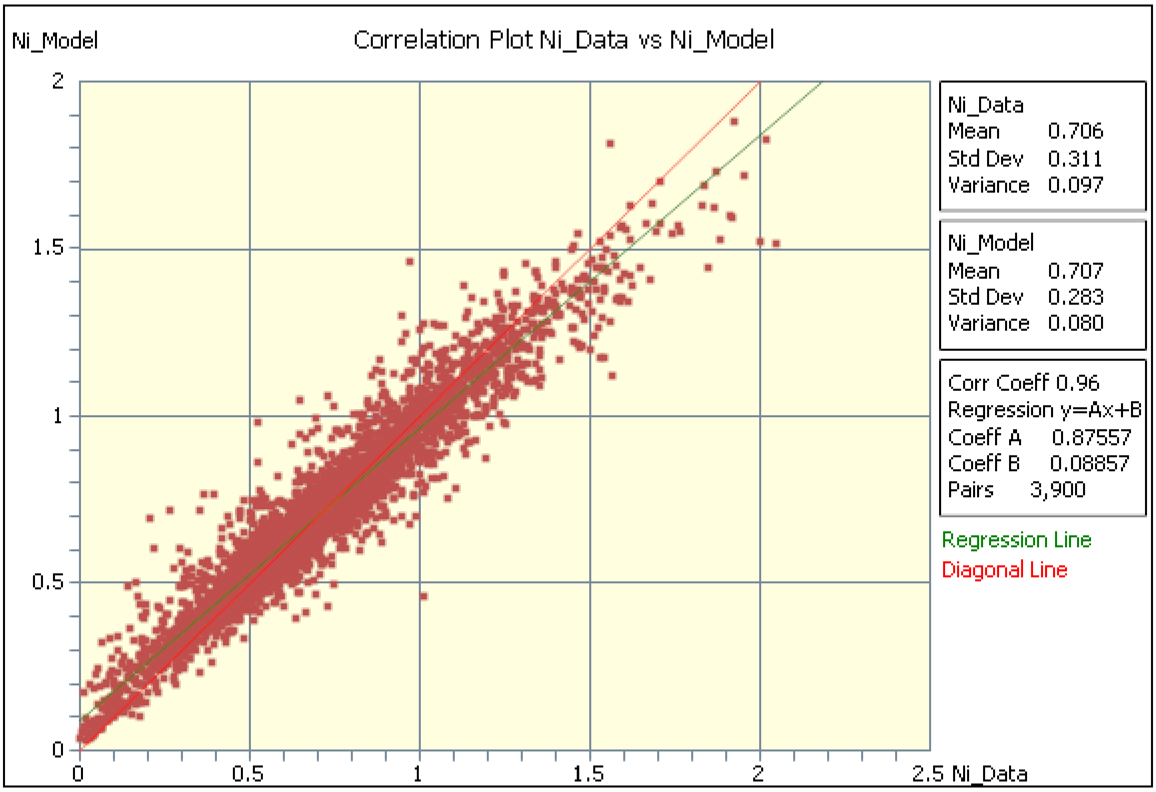


Figure 14-96: Ni GZ

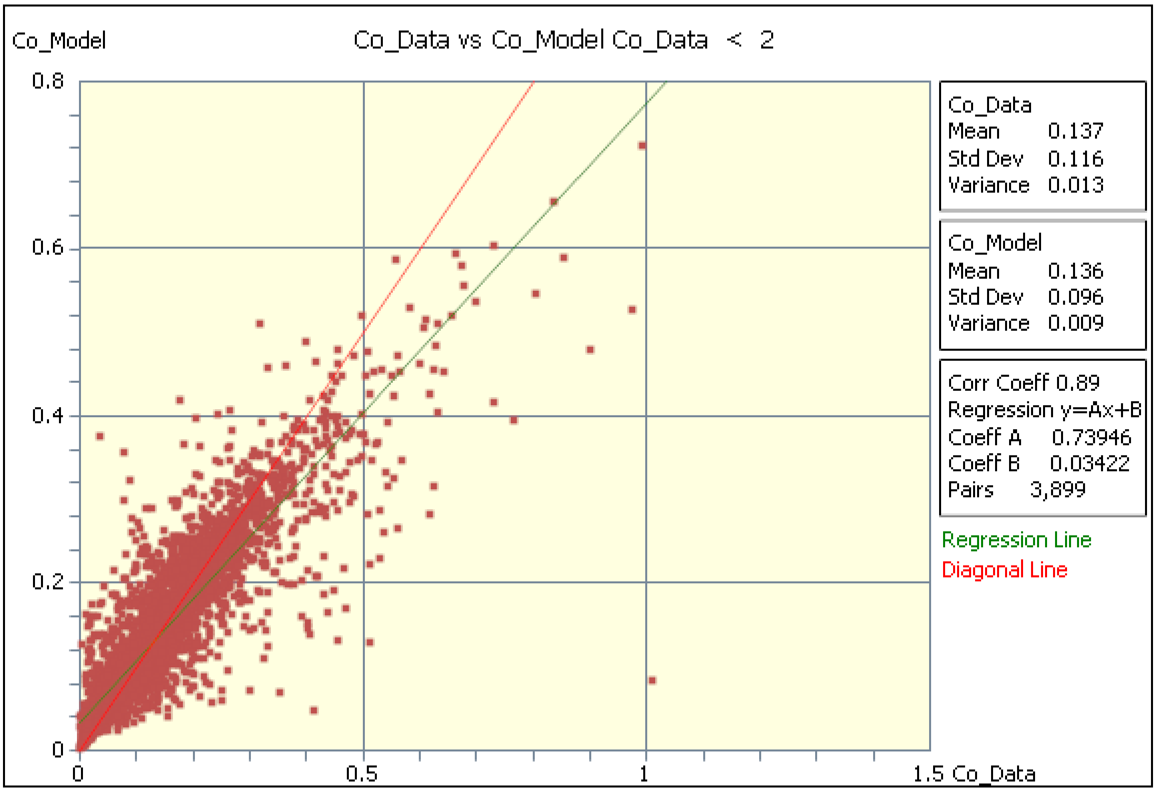


Figure 14-97: Co GZ

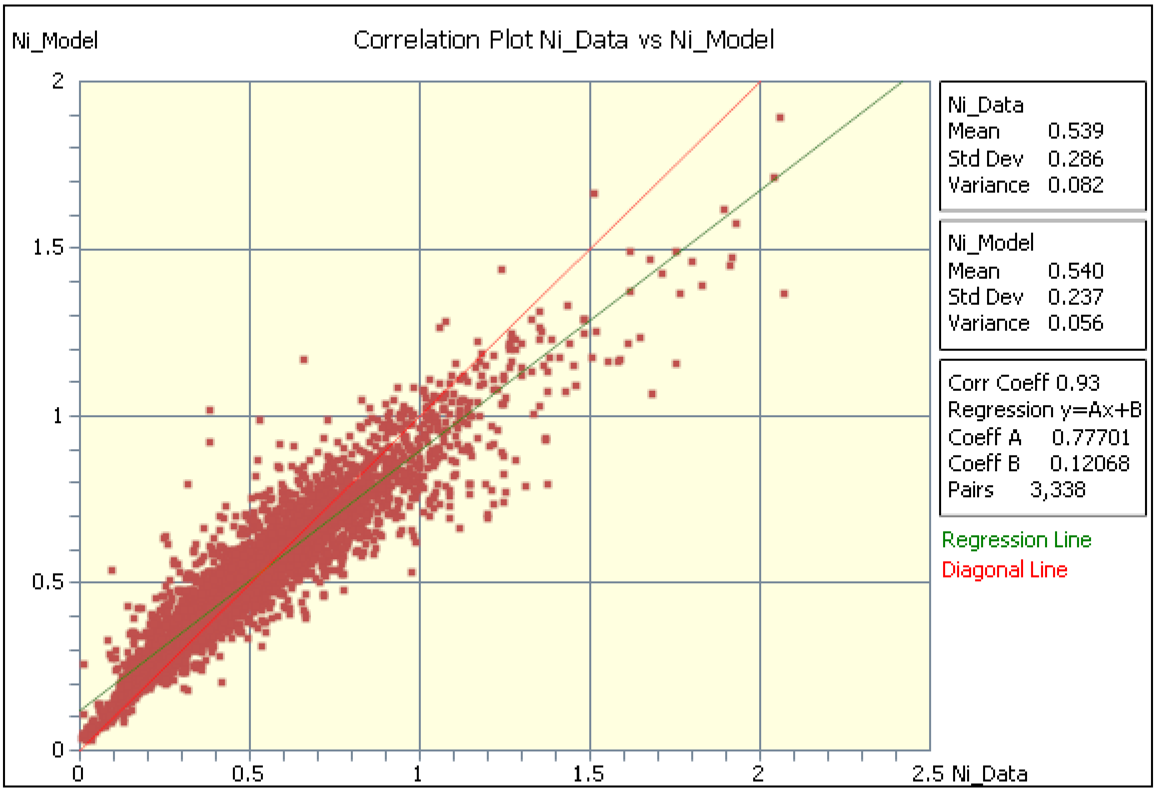


Figure 14-98: Ni SGZ

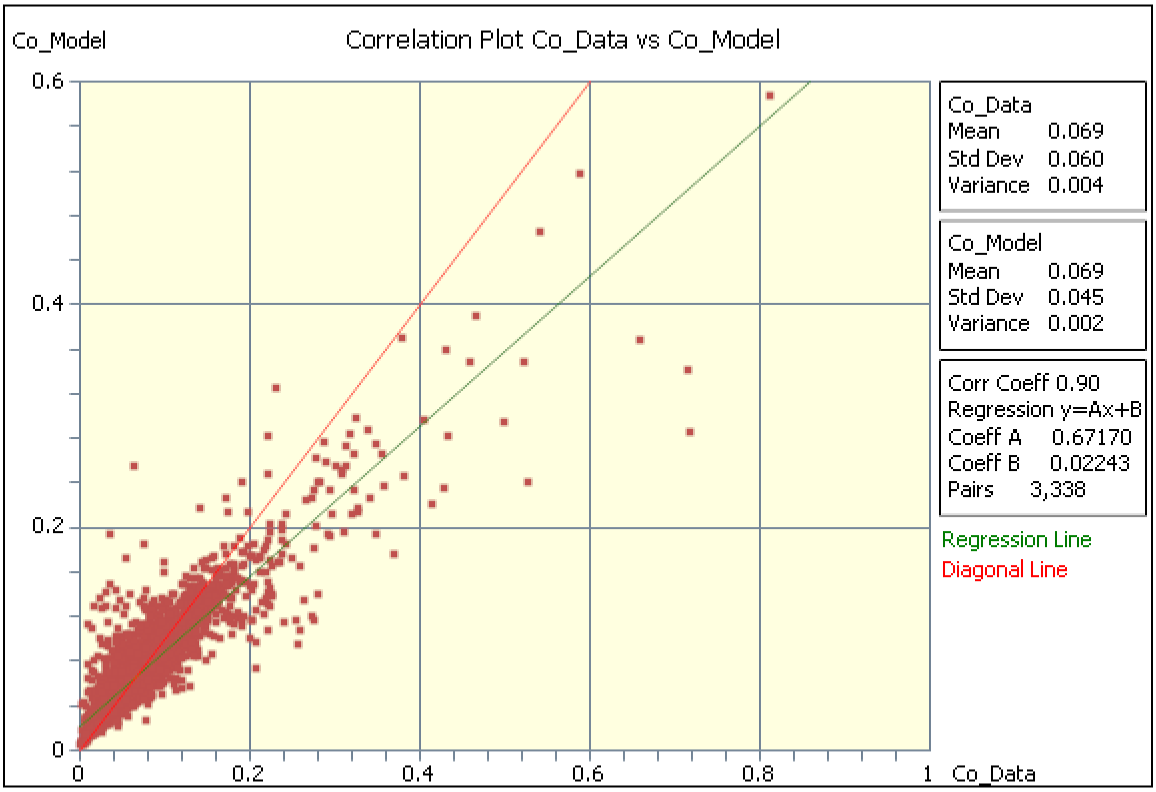


Figure 14-99: Co SGZ

14.3 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 Mineral Reserves (JORC Code). This is effectively equivalent to the Canadian Institute of Mining, Metallurgy and Petroleum 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 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 Clean TeQ 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 rule, the following spacings characterise the Resource Classification:

- Measured: nominal 60x60 m or closer
- Indicated: 120x120 m down to 60x60 m
- Inferred: greater than 120x120 m.

Estimation of major elements has been by Ordinary Kriging. The interpolation has been set up using Kriging Neighbourhood Analysis and testing of the results of various search and kriging parameter combinations. Ordinary Kriging 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 the above, along with the drill spacing and confidence in the geological interpretation, are used to arrive at the final nickel and cobalt resource classification (see Figure 14-100).

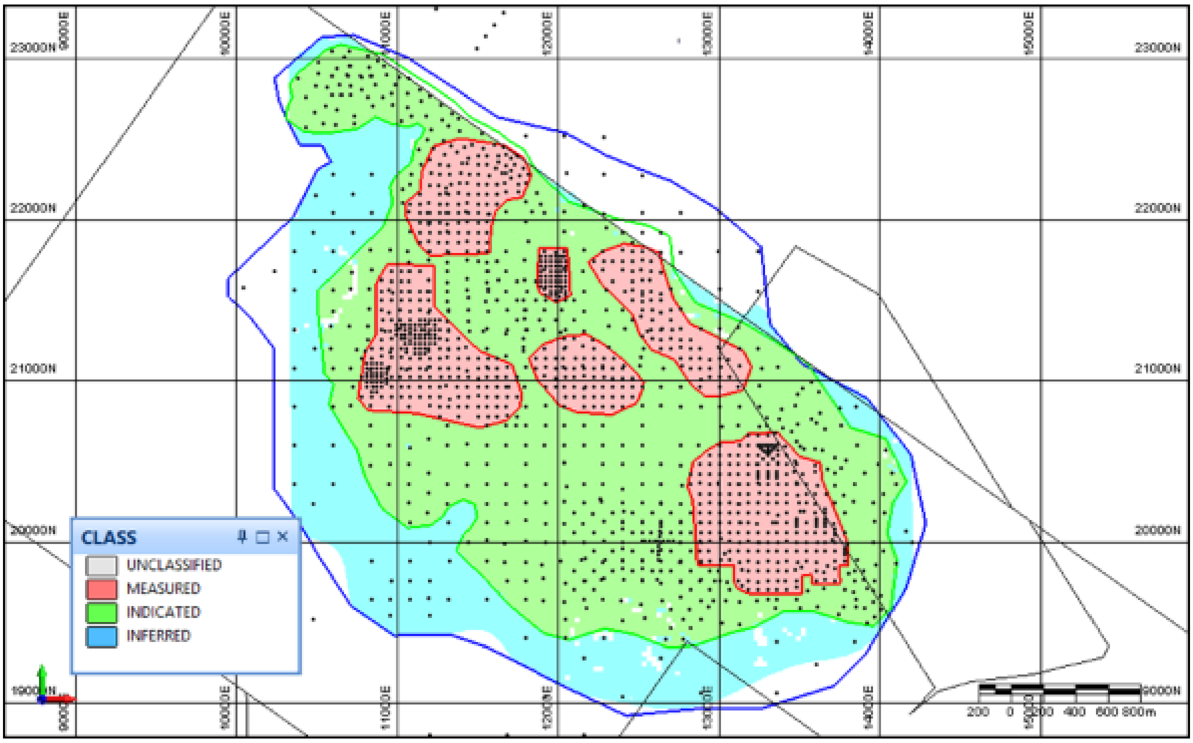


Figure 14-100: Resource Classification Boundaries

Scandium and platinum are 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 in Figure 14-101.

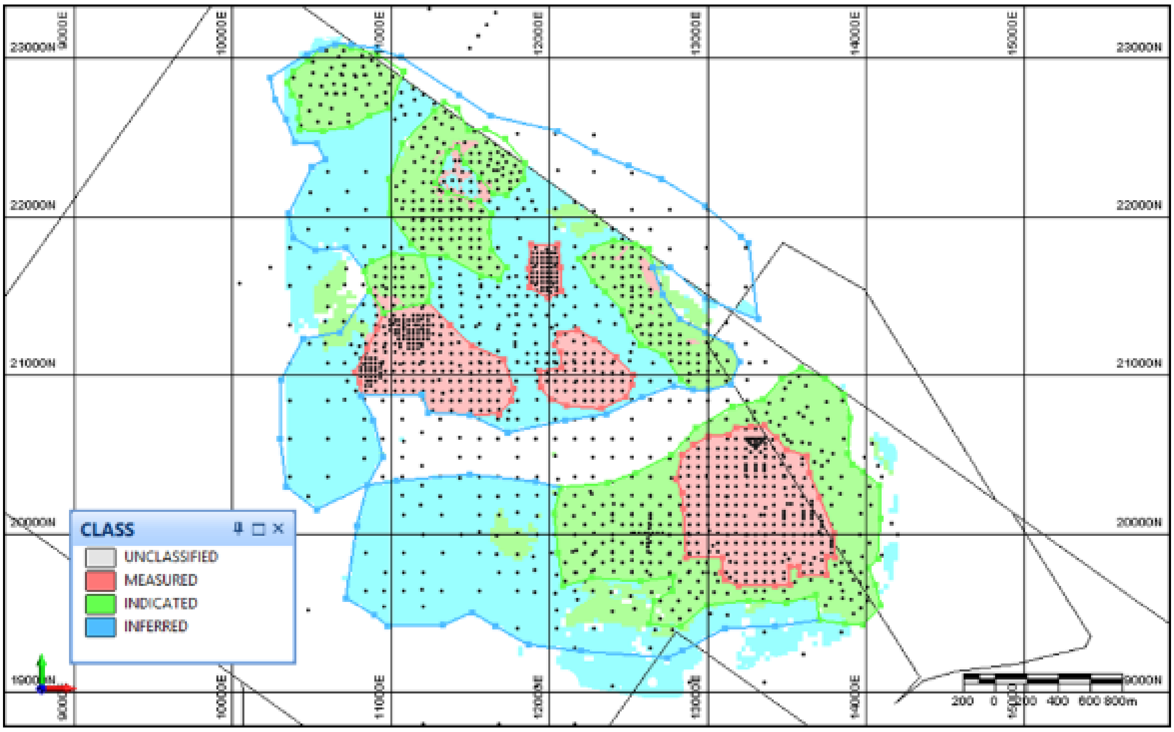


Figure 14-101: Scandium Resource Classification

The platinum data is more limited in all domains and a different set of classification areas are used. 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 as shown in Figure 14-102.

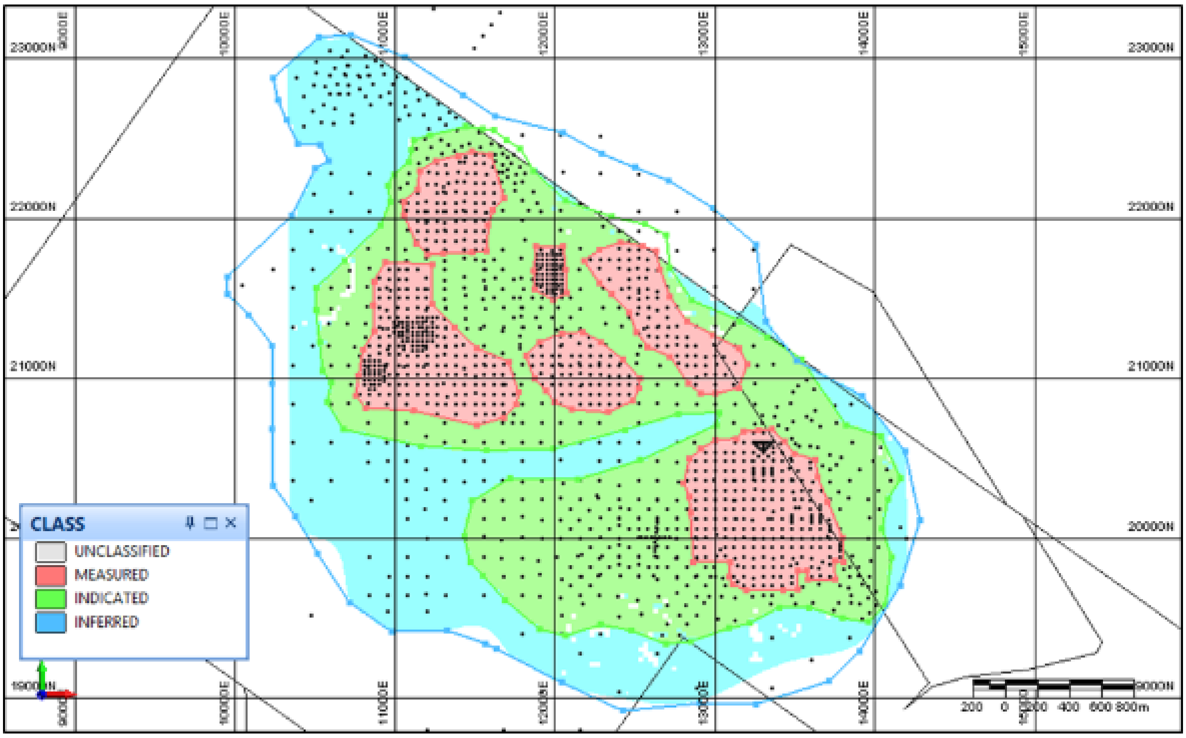


Figure 14-102: Platinum Resource Classification

14.4 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.

14.5 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.

14.6 Metallurgical assumptions

Metallurgical testwork has been carried out on diamond, reverse circulation and sonic core samples from geographically dispersed drill holes, with coverage of all geological domains.

Metallurgical testwork on the nickel, cobalt and platinum material for the Clean TeQ Sunrise 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 the DFS in 2016 through 2018 for mineral recovery determination.

A comprehensive suite of metallurgical test work, including further Pilot Scale test work and specific equipment vendor test work is presently in process and due for completion in Q3 2018 further supporting results used in 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 92% for nickel and 91% for cobalt and 31% for scandium. The metallurgical recoveries for nickel and cobalt were derived from metallurgical test work comprising over 150 ore variability batch tests and four separate pilot plant campaigns testing 10 bulk ore composites as part of three feasibility studies completed in 2000, 2005 and 2018. Recent metallurgical test work undertaken by Clean TeQ confirm these recoveries. and the 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 Clean TeQ Sunrise lateritic nickel, cobalt and scandium mineralisation. The proposed process for nickel, cobalt and scandium 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 solvent extraction separation and purification, prior to crystallisation to produce battery grade nickel and cobalt sulphates. The proposed process for the scandium refining involves precipitation and purification steps of the scandium eluate to produce high purity scandium oxide product.

14.7 Environmental assumptions

An Environmental Impact Statement was prepared in parallel with the 2000 feasibility study and in May 2001 the proposed nickel-cobalt project received Development Consent under the NSW Environmental Planning and Assessment Act. The area in which the deposit occurs does not seem to have any unusual environmental significance.

The previous granting of a Development Consent indicates that there are unlikely to be any insurmountable environmental obstacles, however additional permits and licences would have to be obtained before operations could commence.

As part of the definitive feasibility study, 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.

14.8 2017 Resource Estimate

The cobalt and nickel Mineral Resource is summarised in Table 14-14 to Table 14-17 at a range of cobalt Cut-Off grades.

The scandium and platinum Resources are reported in Table 14-14 to Table 14-24. Note the platinum resource includes a higher-grade zone of 1.7 Mt at 1.87g/t platinum for 103,435oz at a 1g/t platinum cut-off grade.

Table 14-14: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate at 0.06% Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	40	0.15	0.75	59,000	298,942
Indicated	47	0.12	0.55	58,000	259,479
Measured + Indicated	**87**	**0.13**	**0.64**	**116,000**	**558,421**
Inferred	14	0.11	0.24	16,000	34,643

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 14-15: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate - No Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	69	0.10	0.63	69,000	436,000
Indicated	94	0.08	0.47	75,000	438,000
Measured + Indicated	**163**	**0.09**	**0.54**	**144,000**	**874,000**
Inferred	21	0.09	0.23	18,000	48,000

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 14-16: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate at 0.08% Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	29	0.18	0.78	51,000	227,000
Indicated	32	0.15	0.57	47,000	183,000
Measured + Indicated	**-**	**0.16**	**0.67**	**98,000**	**410,000**
Inferred	10	0.13	0.25	13,000	25,000

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 14-17: Clean TeQ Sunrise Cobalt/Nickel Mineral Resource Estimate at 0.10% Cobalt Cut-Off

Classification Category	Quantity (Mt)	Cobalt Grade (%)	Nickel Grade (%)	Cobalt Metal (t)	Nickel Metal (t)
Measured	22	0.20	0.80	44,000	175,000
Indicated	21	0.18	0.59	38,000	126,000
Measured + Indicated	43	0.19	0.70	82,000	302,000
Inferred	7	0.15	0.25	10,000	17,000

Notes:

1) Mineral Resources are stated according to CIM guidelines and include Mineral Reserves.
2) Tonnes are rounded to the nearest million and contained metal to the nearest tonne.
3) Totals may appear different from the sum of their components due to rounding.
4) The Mineral Resource estimation was verified by Lynn Widenbar, MAIG, who is a full-time employee of Widenbar and Associates and is the Qualified Person under NI 43-101 and the Competent Person for the Resource.

Table 14-18: Clean TeQ Sunrise Nickel/Cobalt Mineral Resource Estimate by Domain (0.06% Cobalt Cut-Off)

LATZONE	Class	Quantity (Mt)	Nickel (%)	Cobalt (%)	Iron (%)	Aluminium (%)	Silicon (%)	Magnesium (%)	Manganese (%)	Calcium (%)	Copper (ppm)	Zinc (ppm)	Chromium (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
	Meas + Ind	**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
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
	Meas + Ind	**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
Silicified 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
	Meas + Ind	**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	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
	Meas + Ind	**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

Table 14-19: Clean TeQ Sunrise Nickel/Cobalt Mineral Resource Estimate by Domain (No Cut-Off)

Latzone	Class	Quantity (Mt)	Nickel (%)	Cobalt (%)	Iron (%)	Aluminium (%)	Silicon (%)	Magnesium (%)	Manganese (%)	Calcium (%)	Copper (ppm)	Zinc (ppm)	Chromium (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
	Meas + Ind	**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
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
	Meas + Ind	**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
Silicified 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
	Meas + Ind	**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	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
	Meas + Ind	**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

Table 14-20: Clean TeQ Sunrise Scandium Mineral Resource Estimate by Domain and Classification (300ppm Scandium Cut-Off)

Zone	Resource Type	Quantity (Mt)	Scandium (ppm)	Scandium Metal (t)	Sc_2O_3 (t)
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
Silicified Goethite	Measured	0.44	439	191	293
	Indicated	3.05	401	1,223	1,871
	Inferred	3.19	392	1,252	1,916
Total	**Meas + Ind**	**12.32**	**420**	**5,175**	**7,918**
	Inferred	33.34	421	14,047	21,492

Table 14-21: Clean TeQ Sunrise Scandium MRE by Domain and Classification (No Scandium Cut-Off)

Category	Resource Type	Quantity (Mt)	Scandium (ppm)	Scandium Metal (t)	Sc_2O_3 (t)
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
Silicified 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	**Meas + Ind**	**245.41**	**79**	**5,175**	**7,918**
	Inferred	117.45	196	23,069	35,295

Table 14-22: Clean TeQ Sunrise Scandium MRE – (No Scandium Cut-Off) TZ, GZ and SGZ Only

Scandium Cut-Off	Classification	Quantity (Mt)	Scandium (ppm)	Scandium Metal (t)	Sc_2O_3 (t)
No Cut-Off TZ, GZ and SGZ only.	Measured	68.81	62	4,238	6,500
	Indicated	93.90	86	8,096	12,417
	Meas + Ind	**162.70**	**76**	**12,334**	**18,918**
	Inferred	20.62	283	5,829	8,940

Table 14-23: Clean TeQ Sunrise Scandium Mineral Resource Estimate Breakdown By Location

Domain	Quantity (Mt)	Scandium (ppm)	Scandium Metal (t)	Sc_2O_3 (t)
Within dunite complex (above 300ppm)	0.3	343	100	153
Outside dunite complex (above 300ppm)	45.4	421	19,122	29,256
Total	**45.7**	**421**	**19,222**	**29,409**

Table 14-24: Platinum Mineral Resource Estimate

Platinum Cut-Off	Classification	Quantity (Mt)	Platinum (g/t)	Platinum (Oz)
0.15g/t	Measured	36.68	0.37	431,491
	Indicated	52.33	0.30	509,065
	Meas + Ind	89.01	0.33	940,557
	Inferred	14.08	0.30	135,614
0.5g/t	Measured	5.33	0.94	161,773
	Indicated	5.16	0.70	116,792
	Meas + Ind	**10.49**	**0.83**	**278,565**
	Inferred	1.65	0.79	41,713
1.00g/t	Measured	1.06	2.12	72,507
	Indicated	0.43	1.47	20,269
	Meas + Ind	**1.49**	**1.93**	**92,776**
	Inferred	0.23	1.44	10,745

14.9 Comparison with Previous Estimates

The cobalt grade of the Mineral Resource has increased by 30%. The Mineral Resource is now 101 Mt at 0.13% cobalt for contained cobalt metal of 132,000t. The nickel grade of the resource is 0.59% nickel 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 at 0.10% cobalt and 0.65% nickel 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 at 0.32g/t platinum for 1,076,170oz, using a 0.15g/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 at 0.20g/t for 700,888oz of contained platinum.

The scandium Mineral Resource for the Project has increased significantly to 45.7 Mt at 420ppm scandium for 19,222t 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 at 419ppm scandium for 11,819 contained metal tonnes, using a 300ppm scandium 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:

- lower grade scandium resource overlying the main dunite basement, and included within the main zones of cobalt and nickel mineralisation
- higher grade scandium resource laterally surrounding the main zones of cobalt and nickel mineralisation.

14.10Resource 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 0.4-0.5%) can have very limited lateral continuity. Local infill drilling to 30x30m 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

The Mineral Reserve estimate was developed as of mine planning work that forms part of the 2018 Definitive Feasibility Study, the results of which were realised on 25 June 2018. The Mineral Reserve estimate is an update to that published in 2017. The Mineral Reserve estimate was developed under the 2012 Edition of the JORC Code. CIM recognises “use of foreign codes” including the JORC Code.

The mine planning is supported by the 2017 Mineral Resource, considering only Measured and Indicated confidence classification mineralisation. The QP has considered the inputs to support the mine planning and confirm that economic extraction of the Mineral Resource is feasible with the planned Hydrometallurgical Facility.

The mine plan was revised to support the Mineral Reserve estimate with updated open pit optimisation incorporating accepted product pricing and current costs and operational parameters. The open pit optimisation underpinned revised mine staging, mine designs and mine production scheduling.

15.1 Parameters Relevant to Mine or Pit Designs and Plans

15.2 Non-technical impacts to the Mineral Resource Estimate

The Mineral Resource Estimate has not been materially impacted by environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant issues.

No material obstructions to the project development have been identified and the project is advancing its permitting as discussed in Chapter 4.

15.3 Mining costs

The DFS revisited the mining methodology and identified that mining needs to be focussed on the selective mining approach, the blending needs, the tight grade variability targets, the focus on mining effectiveness rather than mining efficiency. As a consequence, the processes of seeking costs from a RFQ process was considered to not reliably reflect the true mining costs.

An independent estimator was engaged to build a detailed cost model and cost estimate from fundamentals, in the same manner that a mining contractor would adopt to build a cost model and estimate. The cost estimator worked with Clean TeQ to ensure that the pricing model built accurately represented the pit schedule and required mining criteria at time of DFS.

Key aspects of this cost estimate include:

- all-inclusive unit costs – equipment capital, variable operating costs, labour, supervision etc.
- establishment costs and capital
- monthly fixed charge estimate
- costs based on industry standard equipment costs, New South Wales labour costs, annual centroids, maximum and minimum haul distances for each period and suitable equipment productivities
- allowances for ripping and/or drilling and blasting for the lower SGZ
- excludes grade control costs (drilling and assay).

This information was used to build the principal cost model (PCM) and benchmark against comparable operations.

The cost model was used to build the structure of the tender and evaluation documents and was also used in the financial model.

The PFS mining costs used in the DFS optimisation work are higher than that estimated for the financial modelling for the DFS. The result of this disconnect is that the open pit inventory is smaller than it could otherwise be and presents a level of conservatism to the Mineral Reserve estimate.

15.4 Ore loss and mining dilution

The Mineral Resource Model uses variable parent block sizes, a quarter of the supporting drill spacing.

The resource is flat and laterally extensive with the following points of note:

- The deposit lateral ore continuity
- There is broad width to the ore zones on individual benches
- Vertical and lateral ore contacts are predicted 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.

The mine planning study has selected the Selective Mining Unit (SMU) appropriate for the scale of mining and degree of grade control employed. An SMU of 10m(N)x10m(E)x2m(RL) was selected and applied by regularisation to create the mining model by Clean TeQ to consider dilution and mining recovery.

When comparing the regularised mining model (10m x 10m x 2m) to the Mineral Resource Model, the following dilution and ore loss changes were observed:

- 1.8% more tonnes,
- 2.5% nickel grade loss,
- 2.9% cobalt grade loss,
- 0.7% nickel metal loss, and
- 1.1% cobalt metal loss.

15.5 Geotechnical

Prior feasibility study work concluded that the report, Geotechnical Investigation for the Plant Site and Limestone Quarry Sunrise Nickel Project Fifield, NSW – March 2000 was valid. The 2018 DFS revisited the geotechnical analysis from the 2016 PFS and recognized that the nature of the lateritic profile assigns a degree of uncertainty to geotechnical assessment. Poorly defined or poorly understood geotechnical risks could adversely impact the open pit operation. The impact of the risk could result in failures within the pit which could result in reduced ore access, limited access to pit or increased costs if waste volumes increase.

As such, conservative geotechnical wall parameters and slopes have been adopted that largely mitigated the risk at the expense of higher waste volumes. The pits are relatively shallow (approximately 50m deep) and have a large lateral extent, so consequently the conservatively shallow pit wall angles have a limited impact on waste volumes.

The adopted geotechnical slope parameters used in the mine planning as presented in Table 15-1.

Table 15-1: Slope design parameters

Minimum berm width (m)	Maximum batter height (m)	Overall slope angle (°)	Inter-ramp slope angle (˚)
2.0	2.0	45	32

15.6 Open pit optimisation

Open pit optimisation was undertaken on the 2017 mineral 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.

Strategic scheduling was undertaken using the Datamine NPVS / Multimine software (an add-on module to the Datamine NPVS pit optimisation package). The NPVS package was used to account for all mining and metallurgical parameters and constraints in the optimisation, including plant capacity, mining capacity, grade/metallurgical constraints and long-term low-grade stockpile and reclaim strategies.

The optimisation used to support the Mineral Reserve estimate was performed using only the JORC Code Mineral Resource Measured and Indicated classification. Processing (metallurgical) recoveries for the optimisations were based on recovery formulae provided by Clean TeQ, one for nickel, cobalt and scandium. These formulas were calculated into the software on a block-by-block basis, with the recovered nickel and cobalt grades being used as the revenue-generating elements.

Modifying factors and inputs to the mine optimisation and final pit designs are based on the work recently completed in the DFS, with modifying factors including processing recoveries and costs, infrastructure costs, financial parameters, and geotechnical design parameters provided by professionals competent in these areas.

No explicit cut-off grades were applied in the mine planning, the software does not consider blocks with a net block value per tonne basis below zero, hence a cashflow type dynamic cut-off is applied behind the scenes.

The key parameters used as part of the pit optimisation work to determine the pit extents included (but are not limited to):

- an average of 2.5 Mtpa ore into the autoclave for a nominal 25-year mine life with ramp-up constraints, throughput limits of 25,000t of recoverable nickel and 7,000t of recoverable cobalt
- variable ROM ore feed into the processing plant varying from 2.5-3.4 Mtpa of ore based on beneficiation circuit upgrading of ore
- revenue was determined from recovered nickel metal and nickel price assumptions (USD7.5/lb) and recovered cobalt metal and cobalt price assumptions (USD14/lb) with higher cobalt price assumptions used in the financial model
- mining costs used from the pre-feasibility study produced in 2016 were used for initial mining optimization/ scheduling with updated mining costs produced by independent mining cost estimators.
- Initial non-mining costs used based off initial operating cost models with finalisation of non-mining costs generated as part of DFS work undertaken by other professionals and detailed in Section 21 and 22.

SRK notes that the 2018 optimisation was undertaken using lower metal prices than that applied in the financial model (a nickel price of 7.50 USD/lb verses 7.00 USD/lb and cobalt 14/lb USD/lb verses 30.00 USD/lb. Additionally, as noted above the mining costs used in the optimisation are higher than that estimated for DFS. The net outcome is the open pit inventory is conservatively used and the Mineral Reserves presents a level of conservatism.

The outcome of the optimisation process is a pit shell that provides a geometry guide for the subsequent mine design processes.

The parameters used in the optimisation encourage the software to target higher-grade cobalt and nickel feed during initial mining to maximise the revenue and economic value of the project.

15.7 Mine Design

Pit designs were prepared for the 2018 DFS that forms the basis of this report. Waste dumps and stockpiles were designed to accommodate the volumes in the mine schedule.

Ultimate pit designs were developed based on NPVS derived final pit shell assuming conventional selective mining methods utilising excavators and haul trucks coupled to a ROM stockpile. The pit design parameters are outlined in Table 15-2.

Table 15-2: Pit Design Parameters

Parameter	Units	Value	Comment
Ramp width – dual lane	m	12.4	Cat740 articulated 43t trucks.
Ramp width – single lane	m	6.5	Cat740 articulated 43t trucks
Ramp width – Nominal	m	14	Allow for 777D for waste haulage (single) and Cat740 (Dual)
Wall batter angle	degrees	60	Geotechnical considerations
Overall pit angle	degrees	32	Calculated based on batter angle and berm spacing/bench height
Wall berm width	m	2	Geotechnical consultant
Berm spacing	m	2	Geotechnical consultant
Mining bench height	m	2-5	-

15.8 Pit Sequence Options

Pit staging were developed in NPVS/Multimine. The pushbacks were selected by constraining mining to eastern side of deposit followed by the final pit design. These regions would be used to create practical stages within the final pit design.

Mining to the east side of the deposit was targeted initially to:

- avoid the pit dewatering issues in the drainage channel in the middle of the deposit,
- target high-grade mineralisation
- avoid mining the west side of the deposit to minimise the working areas for operability reasons.

The updated pit designs were used as a basis for the strategic scheduling runs in NPVS/ Multimine.

15.9 Mine Production Scheduling

Mining schedules will aim to mine and deliver consistent feed to the plant that falls below the upper limit of acid demand 300 kg/t and above the lower limit of 200 kg/t as falling below this level constrains energy production from the acid plant. Higher-grade cobalt and nickel areas of the deposit will be targeted in the initial years of project life. There will be a range of mining blocks available with the required nickel and cobalt levels to load the RIP circuit within the optimum range.

To maximise grade and consistency of feed material from the mine, ore will be mined on a just in time basis where possible, with the aim of minimising ore re-handle, loss of grade integrity and avoiding the development of large stockpiles on the ROM pad.

Strategic scheduling was completed using NPVS and Multimine software which uses mixed integer-linear programming to find the pit/phase/bench extraction sequence that maximises an objective function, which in most cases is overall project discounted cash flow.

The software allows the preparation of practical schedules, variable mining rate, processing, multiple mining sources and destination planning. The software allows multiple grade target blending to occur from multiple stockpiles to maximise the payable elements and minimise contaminants. The software allows flexibility within the schedule which allows the algorithms and targets to determine the timing and location of material movement. The software was setup to allow pathways from all stockpiles to the processing plant which allows reclaim from any of these stockpiles to achieve the required targets. Additional constraints via pushback (stages) can be applied.

The DFS mine production schedule developed for the project features a higher-grade, and increased mining movement, with throughput into the PAL at 2.5 Mt ore per annum after ramp-up (mining constraint into eastern side of the deposit)

The initial scenarios were used to establish ultimate pit limit extents and for final pit design. Additional scenarios were used to constrain the pit design.

Scheduling constraints and plant ramp-up schedule based are summarised as follows:

- Year One – 1.4 Mtpa feed of PAL Ore Feed into Autoclave
- Year Two – 2.4 Mtpa feed of PAL Ore Feed into Autoclave
- Year Three onwards – 2.5 Mtpa of PAL Ore Feed into Autoclave
- Deferral of Siliceous Goethite processing until Year 4 of operations
- Annualised production limit of 30,000 tonnes of recoverable nickel and cobalt production;
- Annualised production limit of 7,500 tonnes of recoverable cobalt production;
- Annualised production limit of 25,000 tonnes of recoverable nickel production;
- Metallurgy/recoveries as detailed below

The nickel and cobalt recovery assumption assumptions used to generate the mine plan are differentiated by:

- Destination
  - Feed to the Ore prep
  - Feed into the autoclave
- Silicon Content
- Geological zone (LATZONE) TZ, GZ, SGZ
- PAL cRIP Recoveries

Table 15-3: Processing Algorithms and Mass/Metal Recoveries

Recovered Values – Ore Prep Plant Feed reporting to the HPAL Feed using low aspect ratio, low charge milling												
	Mass	**Ni**	**Co**	**Sc**	**Al**	**Ca**	**Cr**	**Fe**	**Mg**	**Mn**	**Si**	**Zn**
Parameter	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**
Gz Si<15%	100	100	100	100	100	100	100	100	100	100	100	100
SGz Si>15<23%	82	94	94	94	94	94	94	94	92	92	68	94
SGz Si>23%	73	93	93	93	92	92	92	92	89	90	62	92

Table 15-4 summarises the processing parameters used within the pit optimisation/mine scheduling and subsequent financial modelling

Table 15-4: Non-Mining Modifying Factors (Processing)

Parameter	Value	Comments
Ore PAL rate year 1 (Mtpa)	1.414	Ramp-up process design
Ore PAL rate year 2 (Mtpa)	2.462	
Ore PAL rate (Mtpa)	2.5	Basis of design
Metal Production Rate		Basis of design
- Total (tpa)	Up to 30,000	
- Cobalt (tpa)	Up to 7,000	
- Nickel (tpa)	Up to 25,000	
PAL cRIP O/A cobalt recovery (Metal R %)	91.5	Mine planning (optimisation)
PAL cRIP O/A cobalt recovery (Metal R %)	91.2	Financial model
PAL cRIP O/A nickel recovery (Metal R %)	93.4	Mine planning (optimisation)
PAL cRIP O/A cobalt recovery (Metal R %)	92.6	Financial model
PAL cRIP Sc_2O_3 recovery (Metal R %)	91.0	Mine planning (optimisation)
PAL cRIP Sc_2O_3 recovery (Mass R %)	50	
PAL cRIP Sc_2O_3 overall recovery (%)	31.5	Financial model
Acid addition (kg/t)	PAL acid consumption (kg H_2SO_4 per tonne (PAL) feed solids) = 24.2 * (Al – 0.72) + 24.5 * Ca + 15.8 * Co + 38.3 * Mg + 10.6 * Mn + 16.0 * Ni + 133.9	Used post reporting acid generated as part of Metsim calculations

Benches were scheduled in such a way as to keep metal production as high as possible, and with the ultimate aim of bringing metal forward. In the initial years of production, 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. Processing of Siliceous Goethite material was deferred to Year 5 of operations with material upgraded prior to HPAL processing utilising processing and metallurgy algorithms as described in Table 15-3.

The mine production over the first ten years averages 15.7 Mtpa peaking at 20 Mtpa to enable accessing of significant high-grade feed to the process plant.

Mining for the first 10 years assumes peak of two 100 t excavators and a fleet of eight 60 t trucks for mining ore and peak of one 250 t excavator and a fleet of eleven 100 t trucks for mining waste/reclaiming from stockpiles. The flexibility in using a mining contractor allows the ability to add or remove machinery as required. Wheel loaders, dozers and additional trucks will service stockpile rehandle as required.

15.10 Schedule summary

Figure 15-1 to Figure 15-4 summaries the DFS mine production schedule. A visual representation of the advancement of the open pit is presented in Figure 15-5 to Figure 15-7.

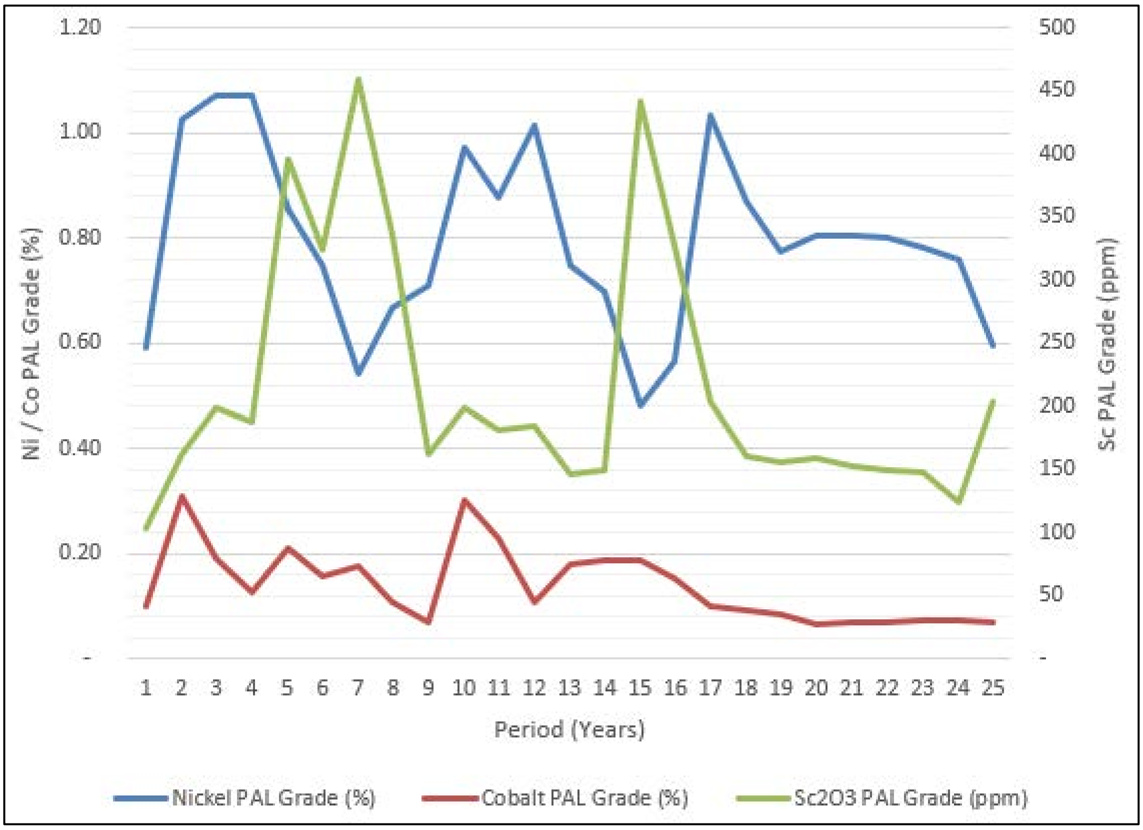


Figure 15-1: Production Schedule DFS Base Case – PAL Feed Grade (Initial Mine Life)

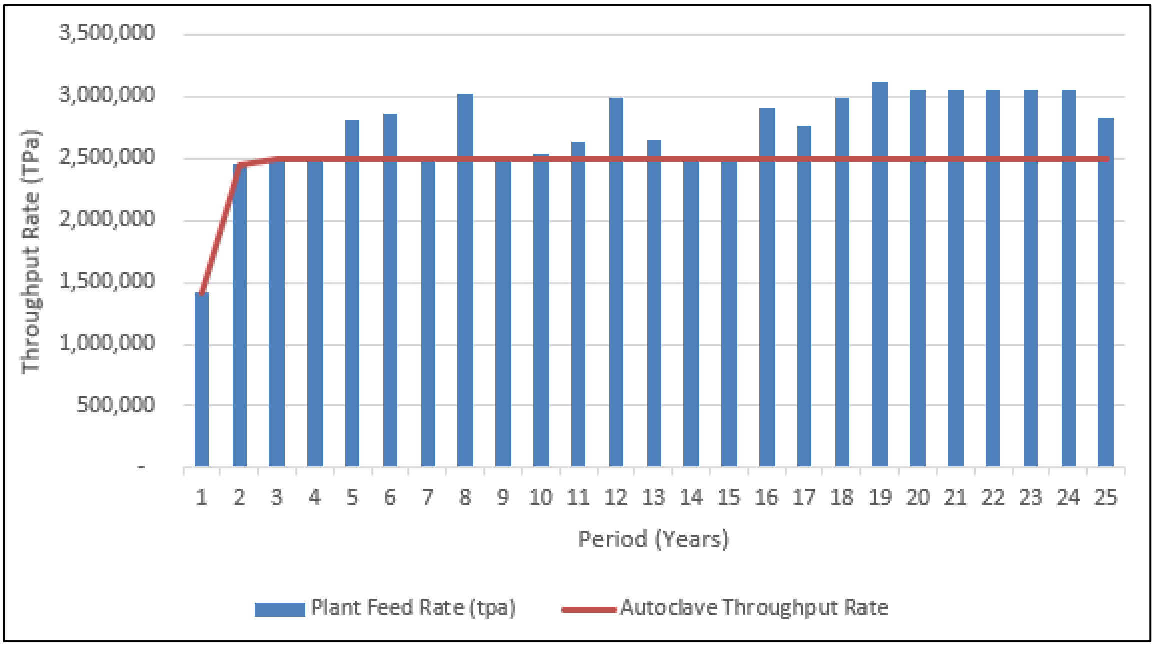


Figure 15-2: Production Schedule DFS Base Case – Annual Throughput Rate Plant and Autoclave (Initial Mine Life)

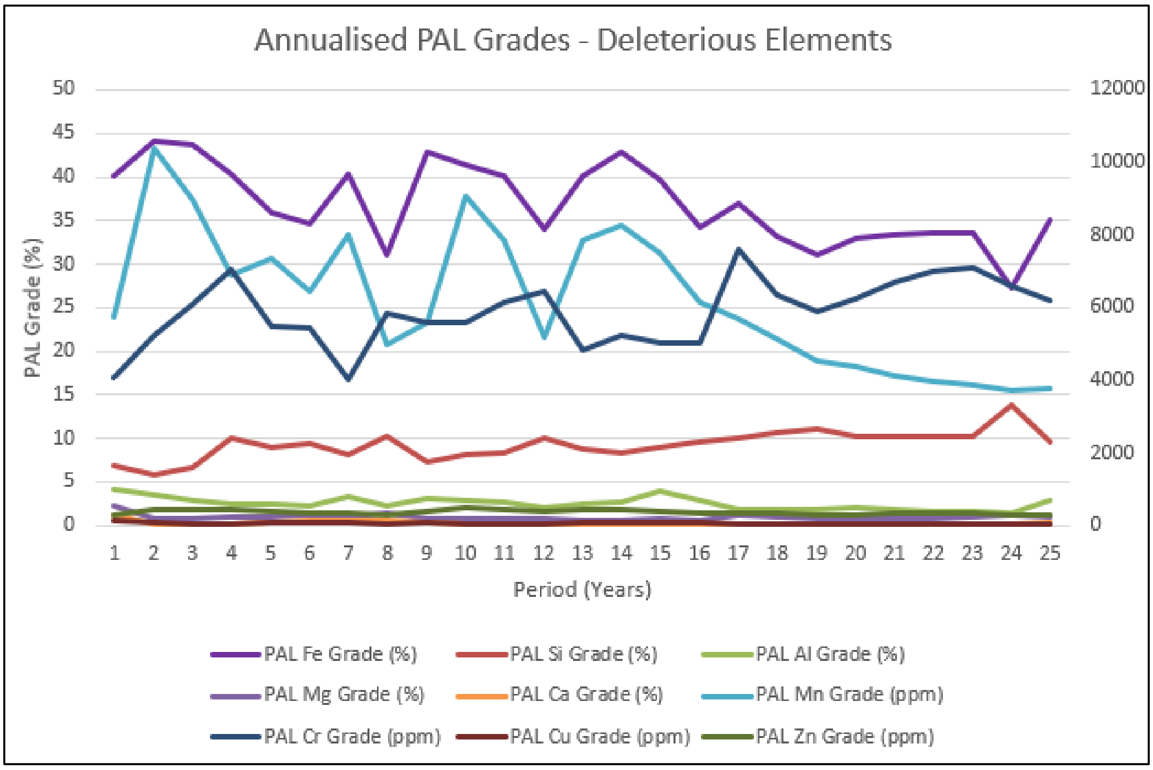


Figure 15-3: Production Schedule DFS Base Case – Annualised Deleterious Elements

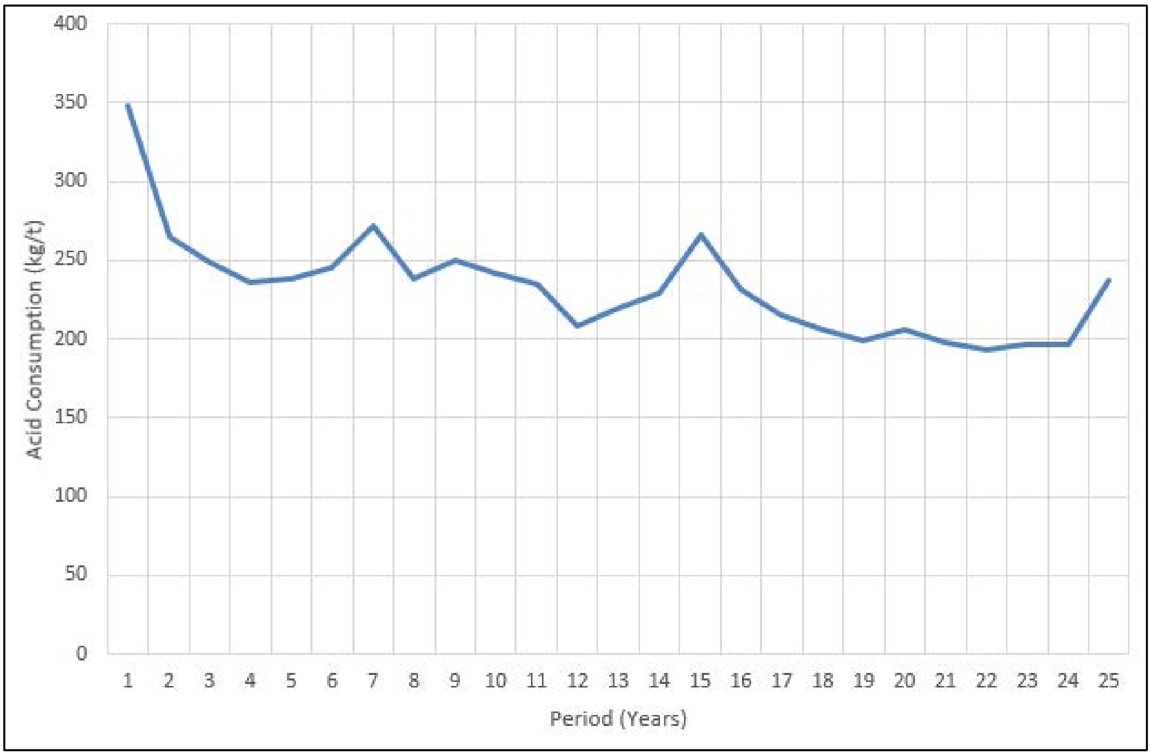


Figure 15-4: Production Schedule DFS Base Case – Annualised Acid Consumption (kg/t)

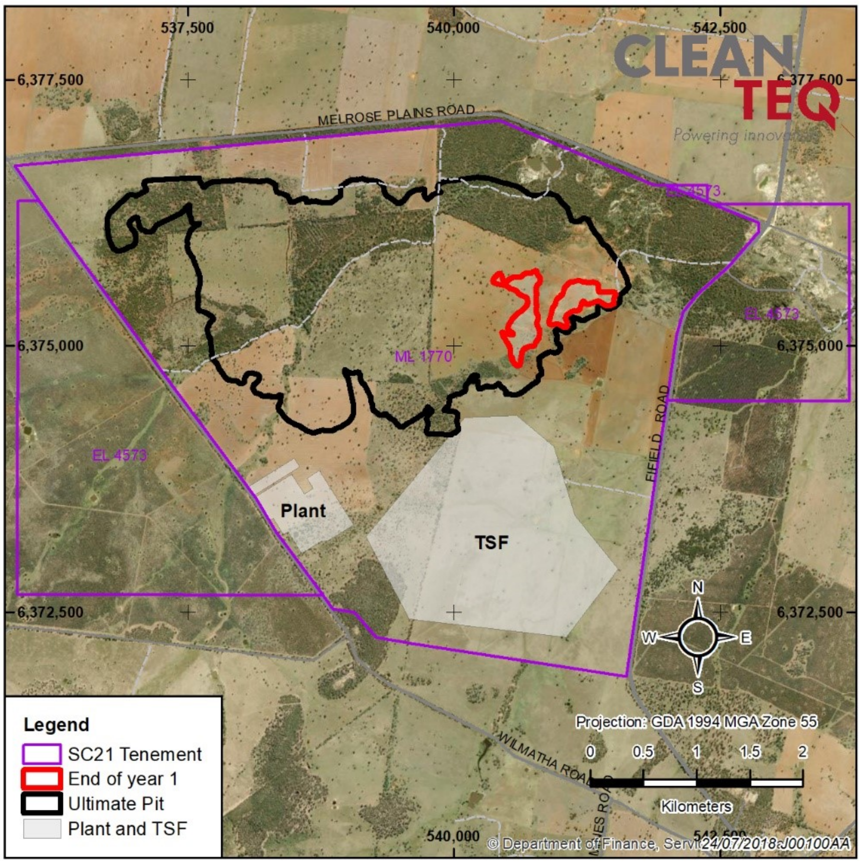


Figure 15-5 Mine Layout – Year 1 / Ultimate Pit

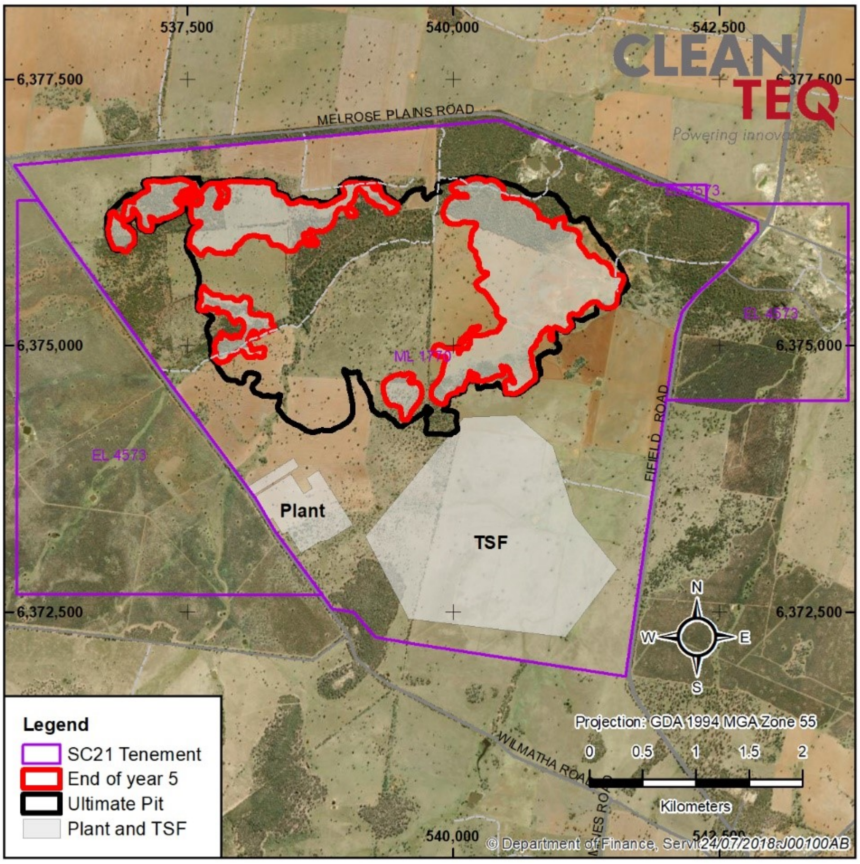


Figure 15-6 Mine Layout – Year 5 / Ultimate Pit

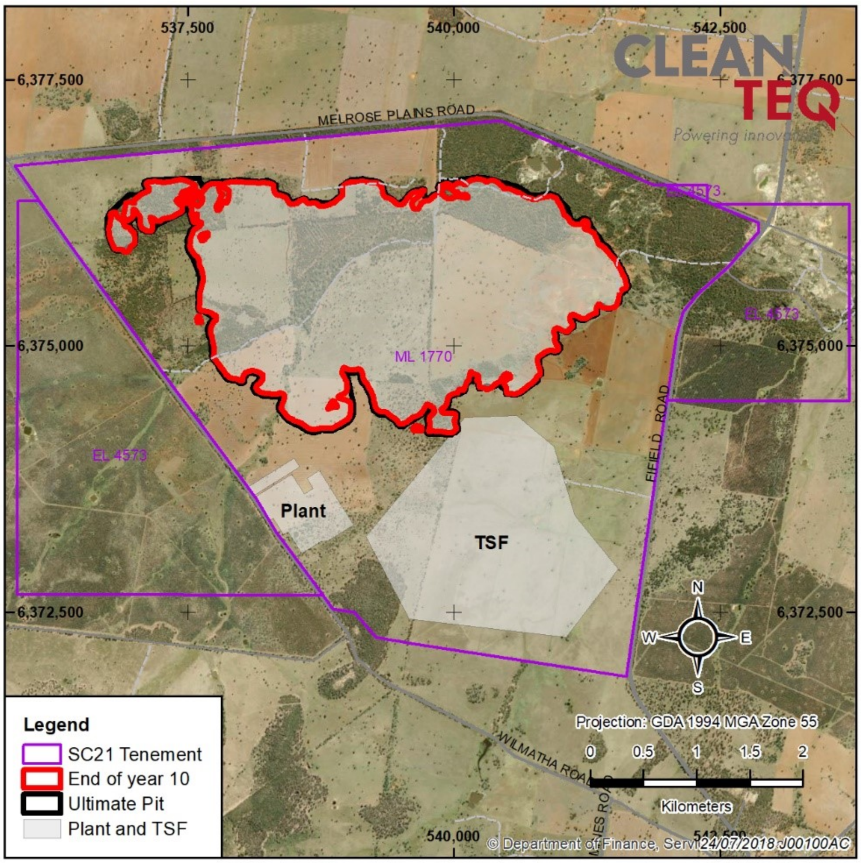


Figure 15-7 Mine Layout – Year 10 / Ultimate Pit

15.11 Mineral Reserve Sensitivity

As part of the DFS, sensitivity analysis was conducted within the Financial Analysis to determine the effect of key variables on the base case post-tax NPV_8 of US$1.39 billion. The results of this analysis are shown in Table 15-5 and Table 15-6 Figure 15-8. Additional sensitivity was undertaken in the initial pit optimization with the DFS Pit Shell based on cobalt price of $14/lb and nickel price of $7.5/lb.

Table 15-5: NPV Sensitivity Analysis

	-15%	-10%	-5%	Base	+5%	+10%	+15%
Capital Cost	1,560	1,504	1,448	1,392	1,336	1,280	1,224
Operating Costs	1,532	1,485	1,439	1,392	1,344	1,297	1,249
Nickel Price	1,082	1,184	1,288	1,392	1,496	1,600	1,704
Cobalt Price	1,156	1,233	1,312	1,392	1,471	1,551	1,630

Table 15-6: NPV Sensitivity Analysis – Metal Recovery

	-4%	-2%	-1%	Base	+1%	+2%
Nickel Recovery	1,302	1,347	1,369	1,392	1,414	1,437
Cobalt Recovery	1,321	1,357	1,374	1,392	1,409	1,427

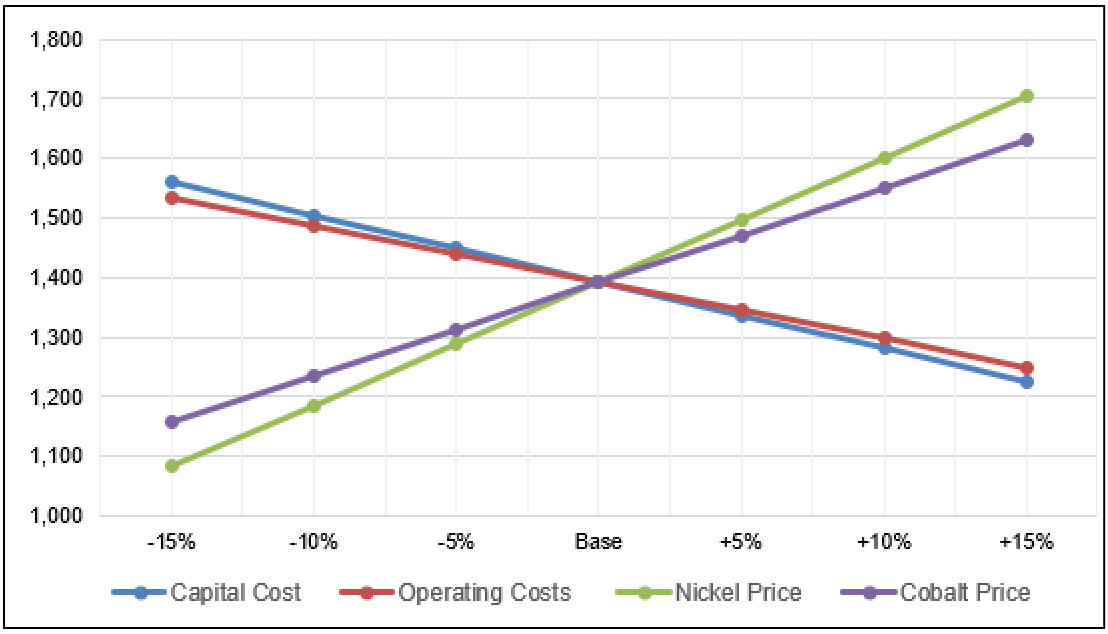


Figure 15-8: Sensitivity Analysis (Post-tax NPV$_8$, real)

15.12 Mineral Reserve Estimate

This Mineral Reserve is the economically mineable part of the Measured and Indicated Resource. It includes mining dilution and allowance for losses in mining. Appropriate assessments and studies have been carried out and include consideration of and modification using realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.

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 Measured and Indicated Resources within the mine plan were converted to the Proven and Probable Mineral Reserves estimate presented in Table 15-7.

The QP 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.

Table 15-7: Mineral Reserves

Classification	Inventory (Mt)	Grade (% Ni)	Grade (% Co)	Grade (ppm Sc)
Proven	65.5	0.645	0.103	48
Probable	81.9	0.493	0.084	57
Proven + Probable	**147.4**	**0.561**	**0.092**	**53**

Notes:

1) Tonnes are rounded to the nearest thousand.
2) Totals may appear different from the sum of their components due to rounding.
3) The Proved Mineral Reserve estimate is based on Mineral Resources classified as Measured, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine. The Probable Mineral Reserve estimate is based on Mineral Resources classified as Indicated, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine.
4) The Mineral Reserve is inclusive of Mineral Resources
5) Only Measured and Indicated resource classifications are considered in the estimate.
6) The Mineral Reserve estimate was independently verified by Scott McEwing, FAusIMM, CP (Mining), who is a full-time employee of SRK Consulting and a Qualified Person under NI 43-101.

16 Mining

16.1 Mining methodology

The approach to mining the Clean TeQ Sunrise deposit has been strongly influenced by the need to avoid pervasive, problematic experiences of analogous nickel-cobalt lateritic projects. Inconsistent ore grade, inconsistent ore quality, and difficult ore handling characteristics are the key issues identified from these comparable projects.

The orebody presents as a shallow bowl covered by alluvium and overburden, with thinner ore intercepts and minimal overburden on the edges. The mining area is quite extensive, covering an area approximately 4 km x 2 km. The pits are shallow, with a maximum depth of approximately 50 m. With this profile, it is proposed to target the higher-value areas of the pit in the early years.

The DFS is focused on optimising cash flow through selective ore mining and, where possible, bulk waste removal. The best outcome for the project will be achieved through minimising variation in feed grades to maximise metal recoveries and minimise acid consumption. Given the criticality for consistent feed grade to the processing plant, it is less important for the mining fleet to be optimised in terms of fleet utilisation and capacity, than it to be flexible and effective in terms of grade control, quality and consistency of mining. The requirement to deliver precise and accurate feed grade and quality to the plant will drive the mining systems, the mining behaviours and the mining outcomes. To this end, the philosophy towards mining has changed in this DFS, from the earlier approach of a contractor bulk mining operation, to that of an owner-managed and highly selective mining operation. It is envisaged the mining fleet will initially be owned, operated and maintained by a contractor on a wet hire/reimbursable type basis with Clean TeQ managing and directing the mining operations.

With the deposit being a flat and shallow basin, it is proposed to mine from multiple working faces, which will provide alternate sources to maintain a constant grade feed to the plant.

It is proposed to pre-strip bulk waste material where possible over the starter pit areas, thereby minimising the strip ratio in subsequent years and enabling a smaller mining fleet to achieve required material movements and focus on more selective mining.

The mine is proposed to operate on a continuous 365 days per year, 24 hours per day basis. Continuous operations will be achieved using hot-seat changes and rolling equipment servicing. Ore will be hauled to the ROM pad and direct dumped into the primary sizer or stockpiled where necessary.

16.2 Mining Development Approach

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 will most likely 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.

16.3 Mining Fleet

The in-pit materials (ore - grade, transitional, overburden and alluvial) are all friable and loose, so underfoot conditions within the pits are expected to be soft, unconsolidated and difficult. In this environment, rigid haul trucks are expected to struggle in wet conditions. Articulated haul trucks are better suited for these floor conditions and are also well matched in size for selective mining and the planned SMU. It is unlikely that sufficient quantities of suitable sheeting material would be available to allow the use of rigid haul trucks for ore and all waste. The stringent grade and quality control targets preclude the use of anything but ore-grade sheeting. Apart from some of the deeper siliceous goethite, very little ore grade material is likely to be suitable sheeting.

Mine operating strategies and equipment selection has centred around selective mining methods utilising excavators, haul trucks and possibly ripping or drilling and blasting methods (if required). Backhoe excavators are proposed as the preferred excavation tool due to their superior ability to achieve selective mining through sitting on top of the material being excavated.

Aligned to the selective and small-scale mining requirements for mining ore, a small-scale fleet of 100 t excavators and 40- 60 t (or less) articulated haul trucks are proposed. This scale of fleet is better matched to the proposed 2 m mining bench heights and offers the flexibility and mobility to quickly move between benches and mining areas to meet mine scheduling requirements and provide a consistent head feed grade.

The waste mining allows larger equipment to be used and a combination of 250 t excavators, and 100 t dump trucks are proposed. The ancillary fleet includes D9/D10 track dozers, 14H/16H graders, water carts, rollers, service and stores trucks, front end loaders and lighting plants, rock breaker, explosives transport and charge up vehicles and light vehicles.

For the purposes of topsoil removal, and for some areas of pre-strip, a contract scraper fleet may be utilised. This will be dependent on the thickness of the layers concerned, the proximity and smoothness of ore/waste boundaries, and the length of haul. Scrapers will most likely be suitable for stripping topsoil for all the ex-pit infrastructure areas, and for some of the outer pit areas. Scrapers can potentially provide several benefits, including lower bulk waste excavation costs, quick and efficient waste and topsoil removal and reducing the remaining strip ratio for the balance of the mine life, thereby facilitating a smaller and more appropriate mining fleet for a deposit of this nature. The viability of using a scraper fleet will be examined during pre-production planning.

16.4 Drill and blast

Whilst drilling and blasting (approval for blasting is dependent on approval of Mod 4) has been included as a phase of the mining operation this is planned to be minimised where operationally possible. The ore zone materials (Transition, Goethite and Siliceous Goethite zones) are loose and friable in nature. Given this, drill core has been difficult to obtain and, consequently, very little rock mass strength data has been obtained. In the absence of this information, the use of empirical measures (geological logs and RQD measurements) is the only viable method of estimating the material strength and likely blasting requirements. The majority of the material is considered to be free-dig material (alluvium, overburden, goethite and the upper siliceous goethite zone). An estimated 3% of the total material movement is estimated to potentially require blasting. This is mainly the deeper siliceous goethite zone that presents in the later years of mining and is consider a low risk to the operations.

Areas of siliceous goethite cannot be free dug will be assessed for ripping prior to consideration of drilling and blasting.

If drilling and blasting is required for the lower siliceous goethite material, for grade variability control, it is planned to drill and blast on nominal 5 m benches. A bench height of 5 m still maintains an adequate blast hole depth to meet blasting parameters. Clean TeQ notes that future investigations into drilling and blasting activities might further optimise bench heights, and blast hole diameters.

The pit optimisation work assumes the open pit material will be free dig with some dozer ripping required in areas of siliceous goethite, this assumption is based off geological core logging, rock quality designation (RQD) measurements, test pits and ore body knowledge. If excavator productivity falls below acceptable levels (consistently) a contingency for drill and blast has been included using low powder factor blasts. The contingency represents 25% of the siliceous goethite rock unit by volume. Grade control is paramount and, in areas where drilling and blasting is deemed to be necessary, the fundamental aim will be to loosen the material with minimal heave or throw. Any remaining oversize material will be broken up with a dozer or, in extreme cases, with a rock breaker.

16.5 Dewatering

High rainfall during wet periods will result in the accumulation of incident rainwater in the pit. Hydrogeological studies indicate that pit dewatering ahead of the mining will not be required. Groundwater flows are insignificant and very little perched water has been identified. One small area of the pit intersects the water table in the lowest benches, however groundwater flows are insignificant. Drainage sumps and pumps will be used to control incident rainwater and will be sufficient to control and mange any groundwater inflows. Additionally, catchment and diversion drains on the perimeter of the pit and at the toe of the waste dumps will be established. Clean water will be allowed to flow through the site via the diversion drains whilst water collected within sediment dams will be collected and pumped to a collection pond for use in mine dust suppression and in the process plant.

16.6 Grade control

The ore and waste materials are expected to be free-dig and, as such, there will be no blast hole samples to assist the short-term scheduling and grade control process. Consequently, closely-spaced RC drilling is the preferred method for grade control. A trial grade control drilling program is planned early in the pre-construction schedule as a precursor to pre-production grade control drilling. The purpose of the trial is to validate the proposed drill spacing and sampling frequency and assist the mine scheduling process and grade control strategy.

RC grade control drilling will be undertaken to a depth of at least 20 m where possible, using 1 m samples. All material, ore, low grade and waste, will undergo grade control drilling to establish greater confidence in the highly variable orebody. The actual drill hole spacing is yet to be determined, however, it is planned that operations will commence with a reasonably close-spaced pattern, which will be reviewed once a greater geological understanding of the deposit is obtained (ie delineation of ore and waste zones and boundaries, and grade variability).

Initially it is assumed that an equidistant burden and spacing pattern of 10mx10m will be used and that grade control drilling and assay will be completed at least three months ahead of the short-term scheduling process and at least six months ahead of actual mining.

All sample prep and assay work will initially be conducted by the site-based assay laboratory.

It is assumed that the onsite laboratory will be commissioned to coincide with the commencement of full pre-production grade control drilling. Grade control drilling will commence in advance of the mine pre-strip activities**.** Assuming that much of the site construction material will be generated by the mine pre-strip activities, the construction and commissioning of the laboratory will be an early stage construction task and is likely to be critical path.

16.7 Waste dumps

Medium to low grade cobalt and nickel material and mineralised waste is proposed to be stockpiled separately from other mine waste so it can be retrieved and processed in the future.

Mine waste is planned to be stored in waste dumps and, where possible, in in-pit waste landforms and used where appropriate to construct tailings dams and bulk earthwork requirements for processing and infrastructure civil construction works.

The planned waste dump capacity of approximately 120 Mt is insufficient for the total waste production of 165 Mt. The 120 Mt capacity will be exceeded in year 15 of the project. Similarly, the planned medium to low grade ore stockpile capacity available is approximately 30 Mt, peaking at 93 Mt. The 30 Mt capacity will be exceeded in year 11 of the project.

Various options exist to increase the available capacity within ten years of project start-up and Clean TeQ is confident that one, or a combination, of the options will be successfully negotiated to solve this lack of capacity within the ten-year timeframe.

Testwork on the waste material has determined that it is non-acid forming and as such, no special management or storage processes have been proposed.

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 Limestone Supply

Limestone feed is required for acid neutralisation at the plant. The Project acid neutralisation design and operating costs are based on a delivered limestone solids grade of 90% CaCO3.

There are a number of potentially viable sources of supply of Limestone in the region to the Sunrise Project. The most attractive sources are presented in Table 16-1.

Table 16-1: Sources of Limestone:

Supplier	$CaCO_3$ %	Estimated Tonnage p.a.	Distance km
Westella	43%	790,000	20
Westlime	95%	416,421	70
Ezylime	95%	416,421	70
Cudal	95%	416,421	140
Molong	95%	416,421	150

The current development consent for Westella allows for 790,000 tpa.

The Westella limestone deposit is located approximately 20 km to the southeast of the proposed Mine and Processing Facility. The design work undertaken in the 2005 Study for mining limestone from Westella was used for the 2016 PFS.

As part of the DFS, Ezylime limestone deposit was identified as a higher quality deposit but further distance from the project with benefits over the additional haulage distance, such as reducing the tonnes of annual production required / reduced size of processing equipment and tailing facilities associated with the lower quality limestone.

At Ezylime, high grade limestone would be crushed and stockpiled at the quarry ready for transport by a transport contractor. An MOU for supply of limestone from the Ezylime operator is in place with CleanTeq to supply limestone for the project.

It was identified that there would be sufficient high-grade limestone feed in the current model for approximately 17 years. Clean TeQ has indicated that the resource continues to the north, but further drilling is required to prove this.

17 Recovery Methods

17.1 Process development

The Clean TeQ Sunrise Nickel Cobalt Project has been in development over an extended period with the original Preliminary Feasibility Study (PFS) completed in 1998 by Fluor Daniel Pty Ltd, followed up by a Feasibility Study (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. Clean TeQ acquired the project in November 2014, and completed a PFS in October 2016, and a Definitive Feasibility Study (DFS) in June 2018, leveraging the previous detailed engineering deliverables.

Clean TeQ has been developing a nickel, cobalt and scandium continuous Resin in Pulp (cRIP) 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 demonstration plant operating on such process streams. A separate FS for a small-scale scandium project at Sunrise was completed in August 2016.

Since late 2016 Clean TeQ has focused on the development of the separate and much larger nickel, cobalt and scandium operation, which is Clean TeQ’s development priority and the base case assumption of the current DFS. During the DFS, Clean TeQ have further developed the cRIP and downstream processing flowsheet for the production of battery grade nickel and cobalt sulphate salts, together with high purity scandium oxide as a by-product. The project will include an estimated 32% scandium recovery from the nickel cobalt HPAL/cRIP laterite flowsheet.

Commercial HPAL processing of lateritic nickel/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 in Western Australia. The technology is now into what is often referred to as the fourth generation of HPAL projects and operations, with examples of previous generation projects being Ravensthorpe, Coral Bay, Ambatovy, Taganito, Goro and Ramu. While some of these operations have experienced materials of construction, start-up and operational issues, the technology is now considered to be well established, and with detailed pre-planning and Project execution these issues can be largely overcome, and the plants should be able to ramp up to name plate capacity in 2-3 years.

17.2 Clean TeQ Sunrise processing plant

17.2.1 Overview

The proposed Clean TeQ Sunrise processing plant is a complex hydrometallurgical processing flowsheet using conventional High-Pressure Acid Leach (HPAL) to leach nickel, cobalt and scandium from the Sunrise ore. The leached nickel, cobalt and scandium is then recovered through continuous Resin-In-Pulp (“cRIP”) and solvent extraction before the final nickel sulphate and cobalt sulphate products are crystallised, dried, packaged and transported to market. A scandium oxide product will also be produced by the extraction of scandium from the nickel cobalt eluate produced in cRIP with scandium liquid ion exchange (LIX), scandium refining and calcination. The final slurry/solutions after metal recovery are neutralised with limestone and sent to a tailings storage facility (“TSF”) or an evaporation pond (for chloride liquors).

The process plant will produce high purity hydrated nickel and cobalt sulphate products, as well as a 99.9% Sc_2O_3 product. Waste ammonium sulphate solutions will be converted to a crystalline ammonium sulphate product (Amsul) for the fertiliser industry. The sales of scandium are capped at 10 tonnes per year in the base case financial model with stockpiling of the remaining production of 70 tonnes per year as an intermediate scandium product. Increased scandium production remains as a possible Project upside.

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 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 nickel laterite hydrometallurgical 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 compared to the use of cRIP including: higher capital and operating costs, larger footprint, as well as lower metal recoveries. The use of cRIP technology addresses many of these issues. cRIP uses solid ion exchange resin beads which are contacted directly with the leached slurry to extract more than 99% 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. No CCD's are required, and downstream solutions are more concentrated due to the concentration of solutions by the ion exchange elution process, meaning that downstream refinery plant size and chemical costs are reduced compared to conventional flowsheets.

The summary flowsheet is shown over page in Figure 17-2 The process steps can be broadly defined as:

- Ore Preparation and Milling
- HPAL
- Nickel/Cobalt RIP
- Scandium Continuous Liquid Ion Exchange (cLX) and Refinery Process
- Nickel/Cobalt Sulphate Purification and Recovery
- Tailings Neutralisation and Storage
- Ammonium Sulphate Crystallisation
- Reagents and utilities (sulphuric acid, steam, water, limestone, lime, ammonia, other).

Further descriptions of each of these key process steps are provided in the following sections.

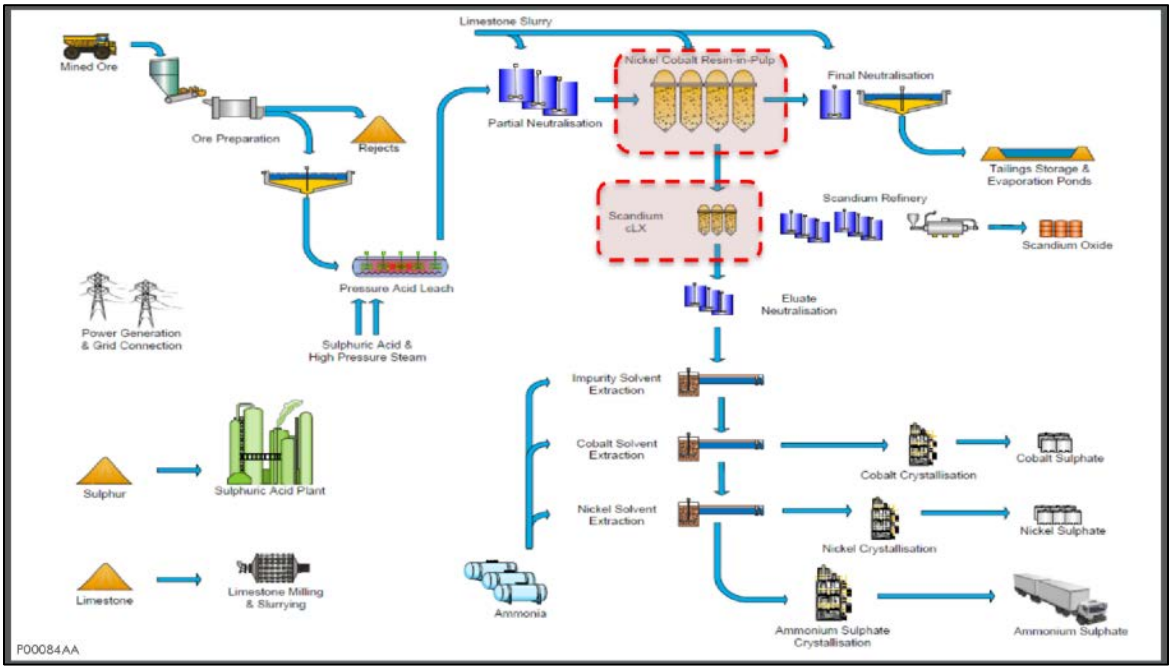


Figure 17-1: Processing flowsheet

17.2.2 Key design criteria

Based on historical data taken from the 2005 Nickel/Cobalt Feasibility Study Update, the 2016 Scandium Feasibility Study, the 2018 Clean TeQ Definitive Feasibility Study, and Clean TeQ's internal database of nickel and cobalt recovery using cRIP, Process Design Criteria were developed for the 2018 DFS.

The operation processes 2.5 Mtpa of feed with a plant availability of 87.5%. Nickel and cobalt leach extractions are 95.7% and 95.0% respectively, and estimated overall recoveries are 92.6% and 91.2% 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 pa.	7,670
Availability Leach Plant	%	87.5
Mine Life	years	25
Average Product Production Post Ramp-Up (Year 2 - 6)		
Ni as Nickel Sulphate ($NiSO_4.6H_2O$)	tpa	21,780
Co as Cobalt Sulphate ($CoSO_4.7H_2O$)	tpa	4,640
Scandium Oxide (Sc_2O_3)	tpa	~ 80
Ammonium Sulphate	tpa	82,000
Nickel Grade (HPAL Feed design)	%	0.985
Cobalt Grade (HPAL Feed design)	%	0.312
Scandium Grade (HPAL Feed design)	ppm	53
Autoclave Operating Temperature	ºC	250
Autoclave Residence Time (minimum)	minutes	79
Sulphuric Acid to Leach	kg/t ore	150 - 350

Estimated Leach Extractions		
Nickel	%	95.7
Cobalt	%	95.0
Scandium	%	91
Estimated PN/cRIP Metal Recovery		
PN/cRIP Metal Recovery (Ni)	%	96.9
PN/cRIP Metal Recovery (Co)	%	96.4
PN/cRIP Metal Recovery (Sc)	%	40.0
Estimated Overall Recoveries		
Nickel	%	92.6
Cobalt	%	91.2
Scandium	%	31.5

The mine production schedule has been derived to ensure target higher grade cobalt is mined mainly in the early years of operation. This is a current opportunity given the present strong market for cobalt.

The process equipment and piping in the process plant flowsheet was sized on the basis of an ‘uptime operating factor’ also known as the On-Stream Time (OST) of 83.8%. In the latter stages of the DFS, Clean TeQ instructed SNC-Lavalin to increase the process plant OST to 87.5% to match the Clean TeQ Sunrise Front End Engineering Design (FEED) OST of 87.5%. However, there was insufficient time to resize the DFS equipment and piping to incorporate the new OST for the DFS within the required timeframe. As a consequence, all equipment and piping remains slightly over-sized at the OST of 83.8%.

A consequence of the client’s instruction to use an OST of 87.5% is that the DFS nominal mass balance changed to reflect the Clean TeQ Sunrise FEED mass balance to accommodate the DFS OST of 87.5%. The new nominal mass balance utilises a dynamic throughput of 45% w/w solids paste thickener underflow (U/F) slurry feeding the pressure acid leach circuit, which is more conservative than the 48% solids slurry used in the original DFS mass balance. This is to ensure dynamic throughput for downstream processing, reagent and utility requirements have been accommodated.

17.2.3 Ore preparation and milling

The purpose of the ore preparation area is to supply the pressure acid leach autoclaves with the tonnage and quality of feed required for high pressure acid leaching of the ore. High level pressure acid leach quality demands consist of delivering high feed slurry density (45-48%), fine particle size distribution (P_{100} of 500µm) and slurry temperature of 84°C.

The orebody essentially consists of heavily weathered goethite that has been impregnated with liberated or loosely bound silica as quartz. The metal values are concentrated in the goethite, which is typically fine or comprised of weakly-bound agglomerates. The silica is relatively barren.

ROM ore passes over a 600 mm square aperture grizzly into the ROM bin, which discharges via an apron feeder to sizer, resulting in a product size P_{100} of 150 mm. The crushed ore is conveyed to a ball mill. The ball mill trommel undersize is discharged to the rejects screen, and the undersize of this screening is pumped to the product screen. The oversize of both ball mill trommel and rejects screen are transferred to the rejects stockpile. The oversize of product screen is gravity fed back to the ball mill while the undersize is pumped to the ore preparation blending tank. The contents of this tank are distributed into two feed thickeners to produce hot dense slurry.

17.2.3.1 Sulphur Dosing

The densified slurry is stored in the pressure acid leach feed tanks, which have a combined capacity of 18 hours production equivalent. Each pressure acid leach train is fed by two centrifugal pumps in series. Sulphur slurry (for addition to the autoclaves as a reductant) is fed to the suction of these pumps.

ROM ore contains localised occurrences of manganese(IV) in pyrolusite-type minerals (MnO_2). Upon leaching in the autoclave, these minerals generate strong oxidising conditions in solution that convert dissolved chromium(III) to chromium (VI). Hexavalent chromium represents an Occupational Health and Safety (OH&S) risk and is harmful to the downstream process. Sulphur is used to reduce the chromium(VI) back to chromium(III).

For ores that generate chromium(VI), sulphur addition to each PAL train is controlled to ensure that approximately 85% of the chromium in PAL discharge liquor exists in the trivalent form. To protect the titanium-lined autoclave vessel from potential corrosion issues, the sulphur slurry is deliberately under-dosed, with supplementary reduction of chromium(VI) to chromium(III) being performed downstream of the autoclave using sodium metabisulphite (SMBS) as the reducing agent.

17.2.4 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, a pressure leach autoclave, and three stages of flash tanks for pressure letdown and steam recovery.

Thickened slurry is heated to approximately 239ºC 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 6,000kPa high pressure 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 compartment one of each autoclave and due to the exothermic reaction, the target operating temperature of 250ºC 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 is re-precipitated before discharging the autoclave. The design of the titanium-clad autoclaves considers materials of construction, sufficient mixing, residence (reaction) time, scaling and corrosion rates.

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 as 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 and alumina
- 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
- Inject sulphuric acid into the first compartment of each autoclave
- Add steam, if required, from the acid plant to the 3rd stage trim heater to maintain autoclave temperature at 250°C; and
- Be able 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 high-pressure seal water supply and vent systems. Most pumps also have a duty and standby arrangement in the HPAL area.

17.2.5 Partial neutralisation (PN)

Neutralisation of the leach liquor to pH range of 4.0-4.2 is required before nickel/cobalt cRIP to allow high selectivity on the resin for nickel and cobalt over impurities such as iron, aluminium and chromium. Moreover, in this process stage, iron, aluminium and chromium are precipitated from the solution prior to cRIP. The leach liquor is also treated with sodium metabisulphite (SMBS) in this stage to reduce any residual chromium(VI), to its much less oxidising trivalent form, chromium(III). This assists in the precipitation of chromium in this stage, and also protects the resins and solvent extractants downstream.

Neutralisation of the acidic liquor occurs with limestone ($CaCO_3$) sourced from a local quarry, producing a precipitate of gypsum ($CaSO_4.2H_2O$) and metal hydroxides. Neutralisation is carried out in a train of six, agitated, partial neutralisation tanks. Fresh limestone slurry is added to the first five tanks. Low-pressure air is sparged into the slurry at the base of each tank to promote oxidation of ferrous to ferric ions, and also strip both heat and gaseous carbon dioxide (generated as a reaction product during the neutralisation process) from the slurry. Neutralised slurry from the final tank is pumped to the mechanically agitated cRIP feed tank.

17.2.6 Nickel /cobalt continuous resin-in-pulp (cRIP)

Following partial neutralisation, the nickel and cobalt contained in the slurry are recovered by the cRIP process which, whilst adopted by other metal processing flowsheets including uranium and gold, and developed for nickel laterites to a pilot level of operation, is not yet fully commercialised for the latter application. The process includes two stages – adsorption and desorption.

To ensure clean separation of loaded resins from slurry, the partially neutralised slurry is screened to remove any oversize material before contacting with resin. The nickel and cobalt contained in the liquor is selectively recovered onto the resin in a 10-stage counter current cRIP circuit using air-agitated Pachuca tanks, with a recovery from solution of over 99%. Limestone addition is used to maintain a slurry pH of 4 in each Pachuca. Pachuca tanks are used instead of propeller-type agitators to minimise the mechanical wear (abrasion and attrition) and breakdown of the resin consistent with commercially operating cRIP plants for both gold and uranium recovery.

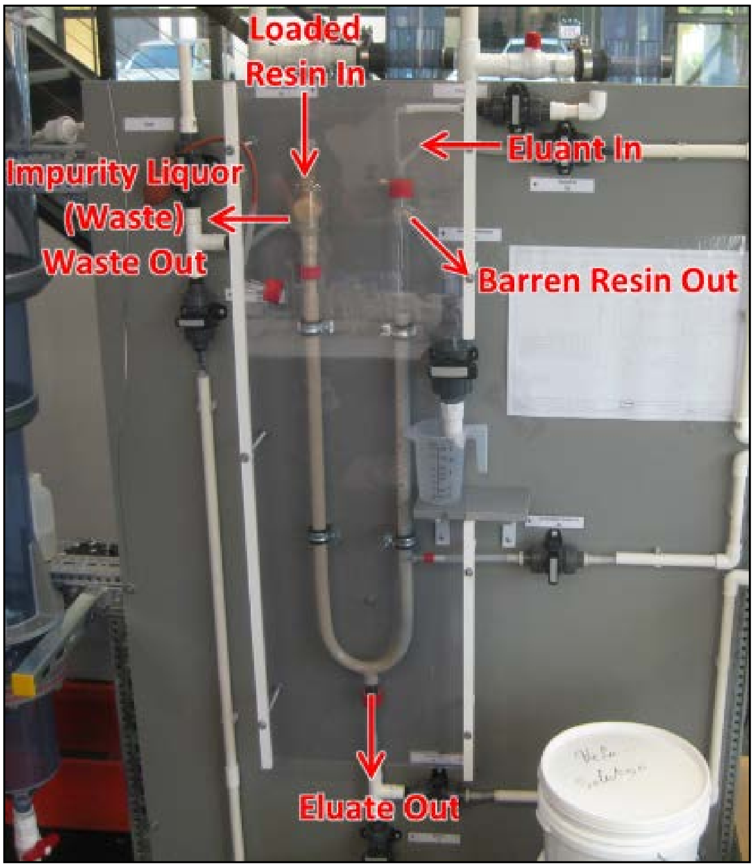
Barren pulp exits the final adsorption Pachuca and passes over two safety screens to ensure recovery of any resin that may have been transferred from the final Pachuca. Screen undersize (barren pulp) is pumped to the tailings neutralisation area for neutralisation with slaked lime, dewatering and disposal. Resin recovered off both resin safety screens is collected in a hopper and transferred back to the cRIP circuit as required.

Loaded resin is screened and washed to remove solids and ultra-fine particles, and then undergoes desorption using weak sulphuric acid, generating a pregnant liquor or eluate (nickel and cobalt sulphate liquor).

The desorption system uses U-shaped continuous counter-current ion exchange columns (Figure 17-2) for a laboratory-scale model) where a small portion of the resin in each column is periodically transferred to the succeeding column, using external airlifts. Desorption solution (eluant) flows down the resin filled leg of the 'U' and the nickel and cobalt are stripped off the resin into the solution. The pregnant solution (containing nickel, cobalt and scandium) accumulates at the bottom of the U-shaped desorption column and is removed as a concentrated nickel/cobalt/scandium liquor product. It advances to scandium continuous liquid ion exchange (Scandium cLX) for scandium removal and refining.

The eluant liquor continues to rise through the resin inlet leg of the U-column, removing impurities from the resin, notably calcium, magnesium, and manganese, and is discharged to the feed return hopper where it is sent back to cRIP feed to recover minor amounts of contained nickel and cobalt.

The barren resin leaving the desorption U-column is washed with water to remove the residual desorption solution. The wash water leaving the column contains an appreciable amount of acid and is used in preparing nickel/cobalt eluant make-up. The barren resin is passed over a screen to remove any undersize and transferred to the nickel/cobalt neutralisation Pachuca where barren resin is partially neutralised with limestone to replace hydrogen ion on the resin with calcium. This ensures that the barren resin recycled back from desorption to adsorption does not disturb the pH level in the nickel/cobalt adsorption Pachucas.



Note: Impurity Liquor = Desorption Waste and Barren Resin = Resin in the H^+-form.

Figure 17-2: Desorption U-column general arrangement

17.2.7 Tailings neutralisation, storage and evaporation

Tailings are neutralised with lime slurry and air addition to remove free acid and precipitate the metal ions as stable hydroxides prior to discharge to the tailings storage facility (TSF). 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 to 42% before being pumped through a single pipeline to the TSF for final storage. The tailings thickener overflow is recycled back to the process water tank.

The proposed TSF is located approximately 500 metres from the processing plant. Tailings disposal will be in three cells, one cell at a time. Tailings are distributed around each cell with spigot take-offs every 10m, 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 pump back any unexpected drainage to the tailings storage facility.

Decant water flows from the TSF to the water transfer pond which takes overflow from TSF during high rainfall periods or as required. Excess solution from the water transfer pond is pumped by the water transfer pontoon pump to the water storage dam.

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.

Streams containing high levels of dissolved chlorides are directed to the chloride evaporation pond.

17.2.8 Scandium continuous liquid ion exchange (cLX) and refining

The scandium continuous liquid ion exchange (cLX) process consists of the two parts - adsorption and desorption.

The nickel/cobalt/scandium eluate from cRIP desorption is pumped to the scandium adsorption column where the resin selectively extracts scandium from the eluate. Resin and eluate flow counter-current to each other. As the solution flows up through the column, the resin is intermittently advanced down the column. The nickel/cobalt eluate then overflows from the top of the column to eluate neutralisation.

The loaded resin slurry is washed to recover entrained nickel/cobalt eluate, and then enters the scandium scrub column, where it is contacted with dilute sulphuric acid to strip impurities from the resin. The scrubbed loaded resin is then washed with raw water before being transferred to the scandium neutralisation Pachuca where the loaded resin is neutralised from pH2 to pH7.5-8.0 (with spent supernatant from the caustic precipitation step in the scandium refinery). The neutralised resin is then screened to separate the neutralised resin from the neutralisation solution. The screened resin then advances to the scandium desorption column.

In the desorption stage, scandium is desorbed from the resin with sodium carbonate solution. The scandium-rich eluate is pumped to the scandium refinery. The desorbed (barren) resin slurry is then regenerated and re-used in the process.

In the scandium refinery, the pregnant scandium eluate solution is batch processed to produce refined scandium oxide. In summary, pregnant scandium eluate liquor is contacted with sodium hydroxide solution to precipitate scandium hydroxide ($Sc(OH)_3$). The solution is filtered and the recovered precipitate either stockpiled as an intermediate scandium product or further processed through a sequential leach, precipitation and washing processes using commercially available reagents. After the final stage of precipitation and washing, the scandium material will be calcined to produce a 99.9% Sc_2O_3 product.

17.2.9 Eluate neutralisation (EN)

Following scandium removal, the nickel/cobalt eluate is heated and then treated with slaked lime in the first three of four agitated neutralisation tanks to raise the pH of the liquor to 4.0 to 4.2 prior to solvent extraction. Low pressure air is sparged into each tank to promote the oxidation of ferrous to ferric ions. Precipitated solids are recycled from downstream to act as seed to promote formation of larger gypsum particles that facilitate more efficient downstream solid/liquid separation. Repulped solids from the downstream polishing filter are also added to the first EN tank.

The slurry is then discharged to the eluate neutralisation clarifier. The clarifier underflow is filtered before sending the solids to the partial neutralisation area. The filtrate is recycled back to the clarifier. The overflow of the clarifier is sent to a polishing filter to remove any suspended solids before proceeding downstream.

17.2.10 Nickel /cobalt sulphate purification and recovery

The neutralised nickel/cobalt eluate is purified and processed in three sequential solvent extraction circuits. The impurity solvent extraction circuit removes zinc, iron, aluminium, manganese and some copper and the raffinate is sent to the cobalt solvent extraction circuit.

The cobalt solvent extraction circuit extracts cobalt, leaving the nickel in its raffinate. The cobalt-bearing organic is sent to a scrub stage to remove impurities and small amounts of nickel before stripping the cobalt from the organic using dilute sulphuric acid.

The cobalt bearing strip liquor is sent to the cobalt purification stage, to remove minor manganese, copper and zinc, before going to a pre-concentrator and a subsequent cobalt sulphate crystalliser to produce high purity hydrated cobalt sulphate ($CoSO_4.7H_2O$). The supernatant from the crystalliser is recycled back to the cobalt stripping stage.

The nickel-bearing raffinate is fed to the nickel solvent extraction step where it is extracted, scrubbed and stripped to produce a concentrated nickel sulphate solution. The product is sent to a nickel sulphate crystalliser to produce high purity hydrated nickel sulphate ($NiSO_4.6H_2O$). The supernatant from the crystalliser is recycled back to nickel solvent extraction, as it contains nickel.

The raffinate from the nickel solvent extraction is sent to the Amsul crystalliser to produce an ammonium sulphate (Amsul) by-product, which is then dried and packaged for the fertiliser market.

The nickel, cobalt and Amsul products are each run through a rotary dryer and stored in product bins prior to packaging. Both the nickel and cobalt sulphate products are packaged in 1 tonne lined bulk bags in separate bagging facilities before being placed in sea containers and sealed with security tags ready for dispatch. Amsul will be stored in sea containers prior to dispatch.

17.2.11 Reagents and utilities

The Clean TeQ Sunrise process plant is a complex hydrometallurgical flowsheet with several reagents and utility requirements. Key reagents and utilities include:

- Sulphuric acid
- Limestone
- Lime
- Ammonia
- Sodium carbonate
- Water
- Power

- Steam.

17.2.11.1 Sulphuric Acid Production and Distribution

Sulphur as prill is delivered via ship to the Port of Newcastle and transported to site, via rail and road trains, where it can be stockpiled or conveyed to the sulphur melting vessel at the required rate.

The sulphur sent to sulphur melting is mixed with hydrated lime to neutralise any acid present in the sulphur or generated in the melting process. It is then melted in a sulphur melting vessel using steam heating coils, mixed with a small amount of filter aid and subsequently filtered. Filtered liquid sulphur is transferred to the sulphuric acid plant where it is burned in a sulphur combustor.

The sulphuric acid plant is based on the latest acid plant technology in which the combustion gas produced contains sulphur dioxide (SO_2), which is transformed to sulphur trioxide (SO_3) in a three-stage converter (utilising a catalyst). SO_3 is then absorbed and transformed into sulphuric acid in the absorption stage. Concentrated sulphuric acid is pumped from the sulphuric acid plant and stored in the sulphuric acid storage tanks.

The acid plant also produces superheated high pressure, saturated high pressure, and saturated medium pressure steam which is exported to the cogeneration plant for power generation and further distribution throughout the plant.

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 when consumption is higher than production. Furthermore, an export facility is also provided to off-load excess acid.

17.2.11.2 Limestone

Limestone is mined by a contractor and delivered to the limestone ROM stockpile where it is crushed to 100% passing 200 mm. The Limestone plant produces limestone slurry at an estimated 35% solids, ground to a P_{80} of 45 µm using a simple impact crusher, overflowing ball mill and hydrocyclone classification system. 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.

17.2.11.3 Other Reagents

There are several other reagents required by the plant. Slaked lime, sodium metabisulphite and sodium carbonate will be sourced by bulk delivery. Ammonia and sodium hydroxide will be trucked in as liquids and stored on site. Hydrogen peroxide (to make up Caro’s acid on site) will be delivered in dangerous goods containers.

Solvent extraction reagents (e.g. D2EHPA, CYANEX 272 and VERSATIC 10) will be stored in Intermediate Bulk Containers (IBCs). Solvent extractant diluent will be delivered in tankers and stored in tanks on site.

Filter aid (in powder form), boiler and water treatment chemicals will be trucked in as solids and stored.

Flocculant will be used in thickener operations in the leach ore feed thickener, tailings thickener and refinery. Flocculant for the leach ore feed thickeners and tailings thickener will be bulk delivered. Refinery flocculant will be delivered in bags.

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 eluate from the desorption circuit of the nickel/cobalt cRIP plant is a high concentration high purity combined nickel/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 lithium ion battery 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

Element	**Unit**	**Target Specification**	**Target Specification**
Ni	% w/w	> 22.0	**< 50 ppm**
Co	% w/w	< 50 ppm	**> 20.8**
Ca	ppm w/w	< 10	**< 20**
Cl	ppm w/w	< 50	**< 50**
Cu	ppm w/w	< 5	**< 10**
Na	ppm w/w	< 20	**< 20**
Mg	ppm w/w	< 10	**< 10**
Ag, Al, As, Cd, Cr, Fe, Hg, K, Mn, Pb, Se, Si, Sn, Zn	ppm w/w	< 5	**≤ 5**
Water insoluble fraction	ppm w/w	< 50	**< 50**
Moisture	% w/w	< 0.5	**< 0.5**

17.3.2 Ammonium sulphate

An ammonium sulphate (Amsul) by-product will be made, primarily targeting the fertiliser market. Amsul products have a mean product size above 1.5mm and a target size range of 40% w/w between 1 and 3 mm and are dried below 1% moisture. The product contains a minimum nitrogen and sulphur concentration of 21% w/w and 24% w/w respectively. Cadmium and mercury impurities are targeted to be below 10 and 5 ppm respectively. Lead will be maintained below 500 ppm. Other nickel laterite operations have marketed some levels of other impurities (e.g. Ni, Co, Mg) as a trace metal component.

17.3.3 Scandium oxide

The process plant includes a scandium circuit that will 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. The forecast initial production is 80 tonnes per year with sales of 10 tonnes per year. More scandium oxide can be produced as scandium demand grows.

17.4 Throughput

From the production schedule for the DFS, the nickel and cobalt tonnage and feed grades are shown below in Figure 17-3. Plant feed tonnage ramps up over two years to the design throughput of 2.5 Mtpa autoclave feed, and then remains at that level throughout the project. Production in Year one is 40% of design (1 Mtpa) and 100% of design by the end of Year 2 (2.5 Mtpa). The feed grade is optimised in the early years of operation, targeting cobalt production, to maximise the Project's returns.

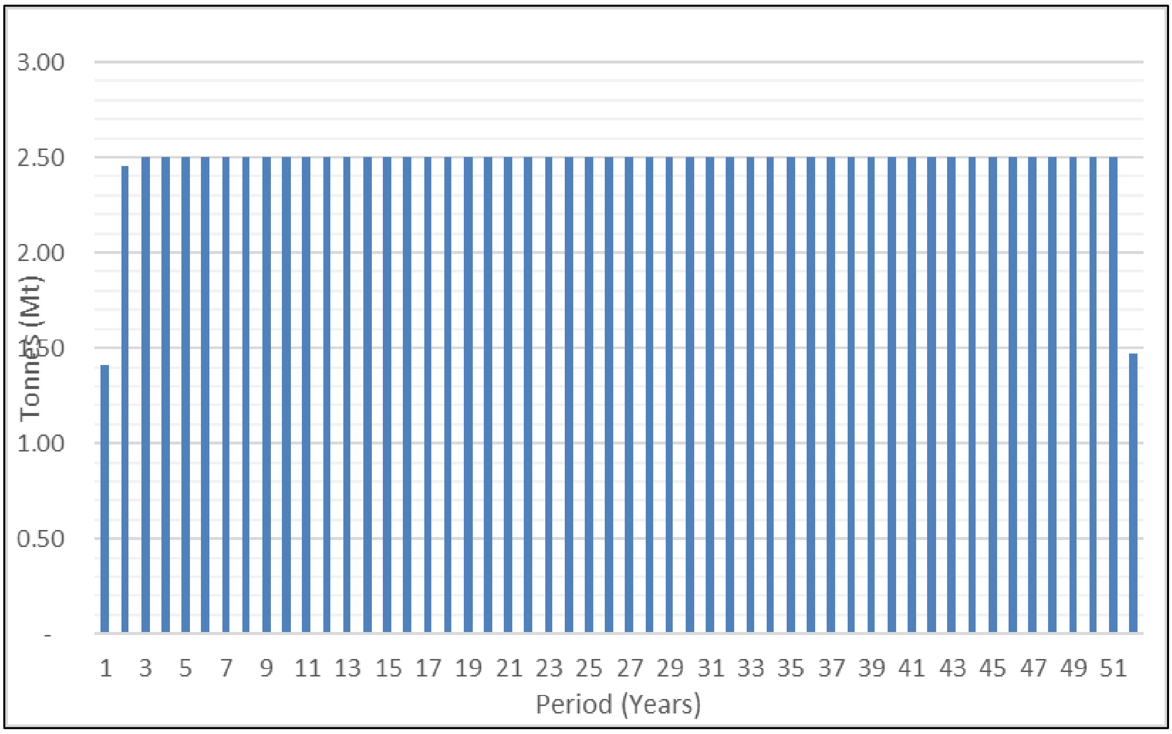


Figure 17-3: Ore Feed into PAL Post Beneficiation (DFS Case)

The associated nickel and cobalt metal equivalent production is shown below in Figure 17-4 together with the amount of mill feed required to produce 2.5 Mtpa autoclave feed.

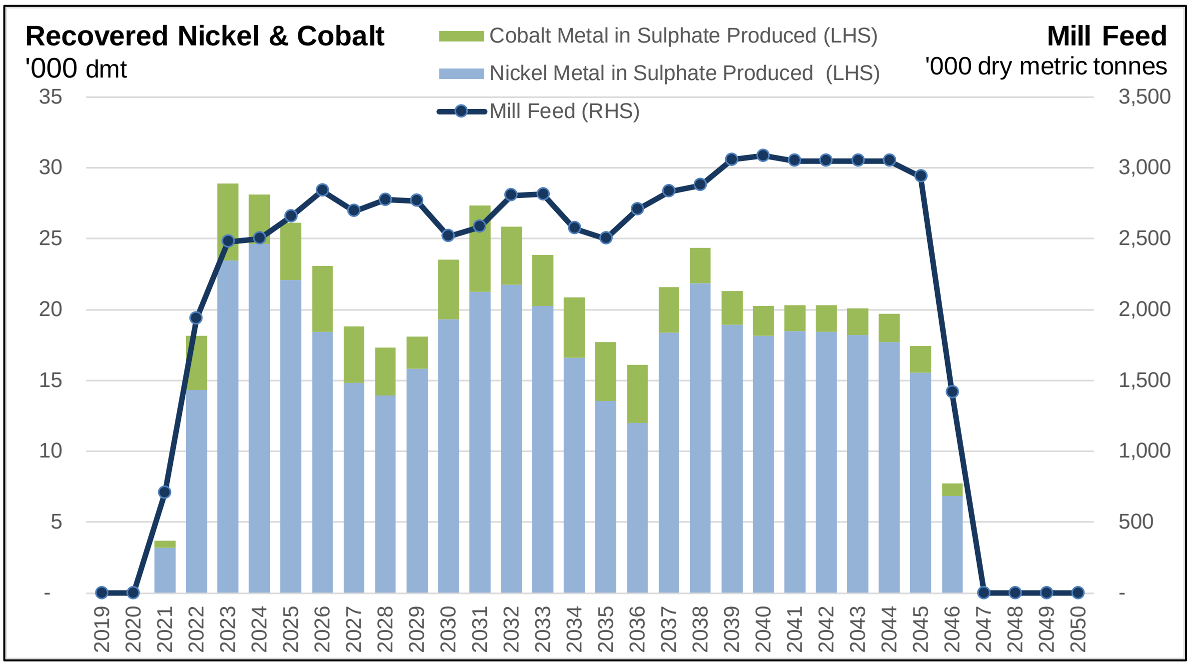


Figure 17-4: Nickel & cobalt metal equivalent production schedule

17.5 Metallurgical recovery

The key processing recoveries are summarised below in Table 17-3. These values are based on the 2005 Feasibility Study, as well as more recent testwork on batch autoclave recoveries, and downstream processing during the 2018 Definitive Feasibility Study. Note that the Feed Preparation circuit recovers 100% to autoclave feed after rejection of the low grade coarse oversize fraction.

Table 17-3: Plant recovery summary

	Nickel	**Cobalt**	**Scandium**
Feed Preparation	100.0	100.0	100.0
Pressure Acid Leach	95.7	95.0	91.0
Partial Neutralisation / Nickel/Cobalt cRIP	96.9	96.4	40.0
Scandium cLX	-	-	99.9
Ni/Co Refinery	99.9	99.6	-
Scandium Refinery	-	-	86.7
Total Plant	92.6	91.2	31.5

Note: Individual stage recoveries, especially partial neutralisation, may also be affected by recycle streams.

18 Project Infrastructure

The Sunrise 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. The Project Location is shown below in Figure 18-1.

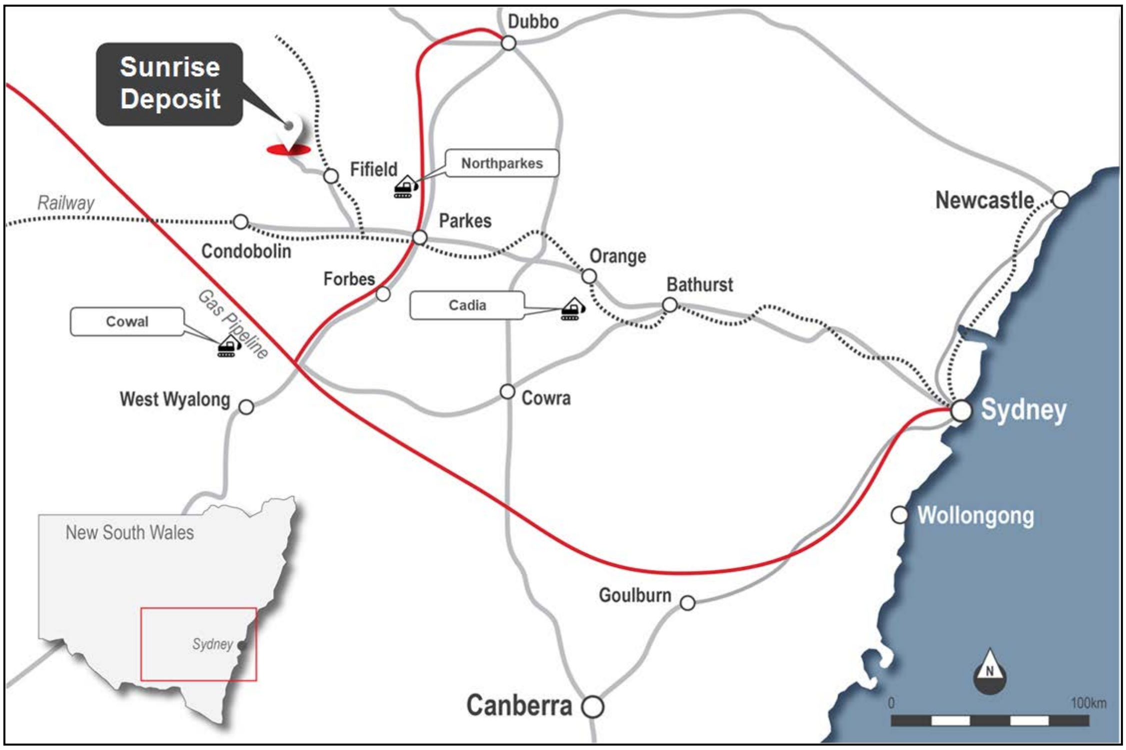


Figure 18-1: Project Location

The proposed Project infrastructure facilities are those required to support a modern 2.5 Mtpa complex hydrometallurgical plant and open pit mining operation. These include:

- on-site and off-site buildings
- raw water supply and treatment facilities
- power from the local grid and on-site power generation
- extension of 4G mobile telecommunication coverage in the area and provision of high speed internet services for the site
- tailings storage facility, evaporation pond, water storage facility, water treatment plant sludge pond and storm water management infrastructure
- fuel storage and transfer
- external road upgrades
- construction camp.

These are discussed in further detail in the following sections. A general mine site layout showing some of the key site infrastructure is shown in Figure 18-2.

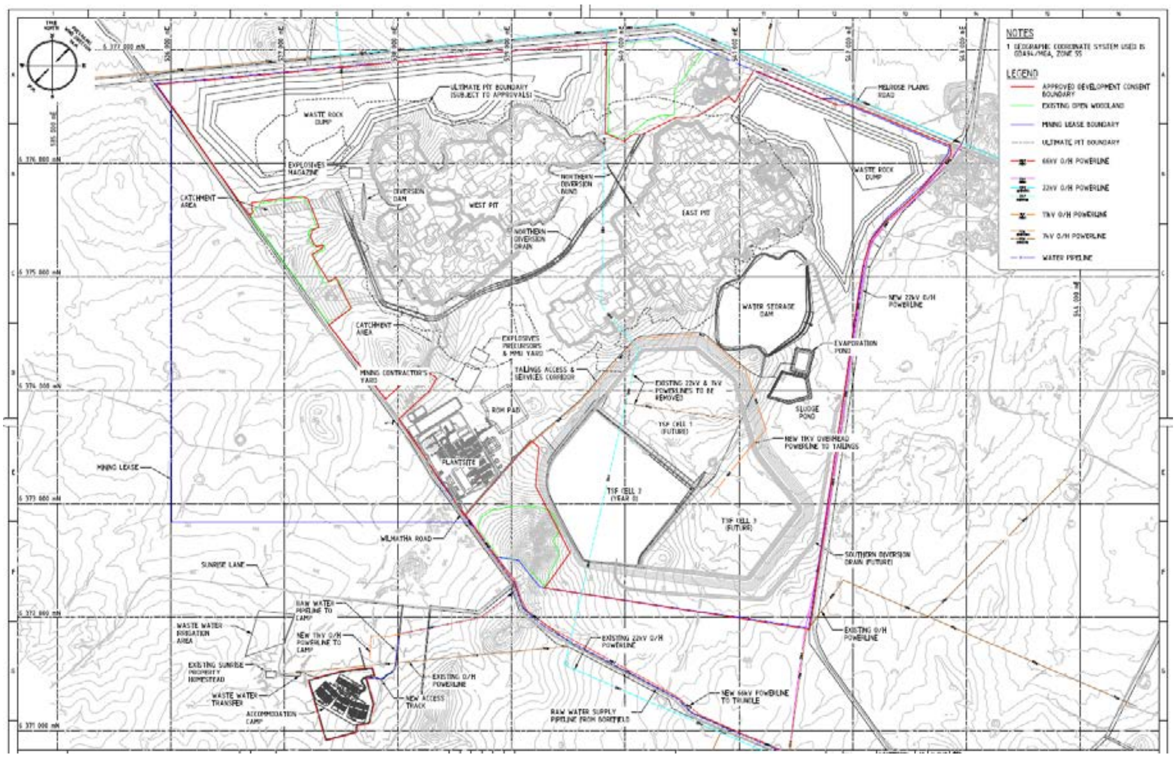


Figure 18-2: Mine site layout

One of the Projects 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, 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 to support project requirements. 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 local population. Several large operating mines exist within the region and the local towns' economies have historically been supported by the regional mining activity. In particular, the former magnesite mine, which forms part of the project area, employed up to 60 local residents. The general response from local government and the community to the re-initiation of the Clean TeQ Sunrise Project has been positive.

The infrastructure design and development has largely been undertaken during the earlier engineering feasibility studies. The work has been continued by Clean TeQ in 2016-2018, specifically associated with the application to modify the Project's 'Development Consent' which has now been given. The Project will make contributions to road upgrades, road maintenance and contributions to community-based activities.

All employees will be encouraged to settle within the central west New South Wales regional towns including Fifield, Trundle, Condobolin, Parkes, Forbes and Tullamore. Bus services will be aligned with rosters and made available to employees to commute from Forbes, Condobolin and Parkes to site. Should employees choose to reside outside the region they will be responsible for their own travel arrangements and costs.

The Clean TeQ Sunrise Construction Camp is located approximately 3 km from site. Following the construction, commissioning and ramp-up, the camp will be cleaned and secured. The camp will be used periodically for shut downs, when it will be required to house up to 250 people.

18.1 Buildings

18.1.1 On-Site Buildings

Four buildings are included in the scope of facilities at the processing plant site. All buildings will be air conditioned with crib and ablution facilities easily accessible. These are:

- site administration and control room building which houses front line operational staff and the site local control room. This includes a local distributed control system console room, local data processing and communication facilities, offices for production, supervision and technical personnel and crib facilities and work stations for production technicians.
- site workshop building which houses maintenance personnel, maintenance work area, tool store, laboratory, goods receival, warehouse space and dispatch area for rotable item spares.
- gate house which includes security, medical and weighbridge facilities
- maintenance contractor facilities with office, lockers and crib facilities sized for shutdowns

18.1.2 Temporary and Off-Site Buildings

During construction, a temporary building of 900 m² will be provided to house Clean TeQ Sunrise advance personnel and process commissioning teams, and a 1,300 person construction camp will accommodate the workforce. After commissioning, the Camp will be reduced to a capacity of 300 people and utilized for maintenance shutdown labour as required.

18.2 Water Supply and Usage

18.2.1 Overview

A reliable supply of water is required for the hydrometallurgical extraction of nickel, cobalt and scandium from laterite ores via a pressure acid leach flowsheet (Figure 18-3).

Clean TeQ will extract water from local bores to provide sufficient water for this process. The water will largely be lost to tailings and evaporation. All required water treatment facilities will be sourced from Clean TeQ Water[23], using their proprietary technology.

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[23] Clean TeQ Water is a 100% owned subsidiary of Clean TeQ Holdings Ltd, an ASX-listed company (ASX:CLQ).

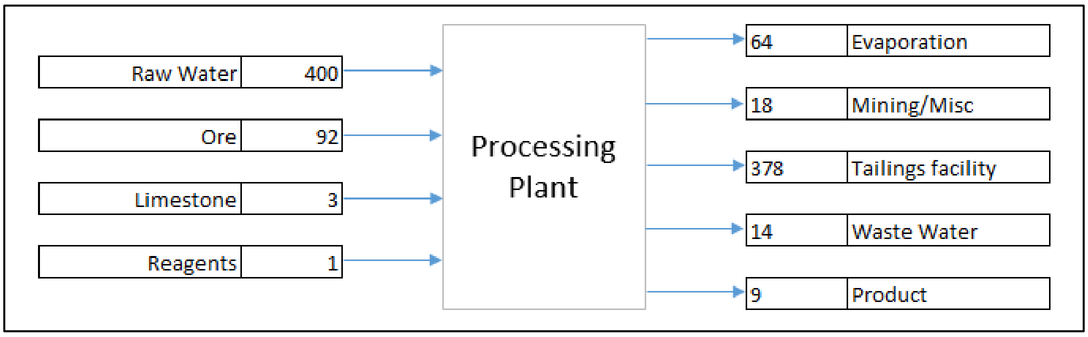

Figure 18-3: Clean TeQ Sunrise Water Balance

18.2.2 Raw Water

Clean TeQ Sunrise will obtain raw water from two borefields adjacent to the Lachlan River, located 70 km south of the processing site. The bore water contains 750 to 850 mg/L total dissolved solids and has a neutral pH. River water will be extracted via a wet well and will be used to supplement the bore water supply when available in abundant supply or it is economically advantageous. The river water contains about 300 mg/L total dissolved solids and also has a neutral pH. It will require filtration to remove suspended solids prior to use.

The plant will receive a nominal 400 m^3/h of raw water (calculated utilising 87.5% plant availability) from the bore fields and/or river via a 70 km long, 450 mm diameter, pipeline constructed from ductile iron, cement lined pipe. The raw water will be received at the plant site into the raw water storage dam, which has a capacity of 30,000 m^3. The estimated raw water requirement is 400 m^3/h during full production. This includes potable water, fire water, high pressure hose down water and mine utility water.

The raw water will be treated on site to produce filtered water, reverse osmosis (RO) water, demineralised water, cooling water and potable water for a range of duties in the processing plant and refineries.

18.2.3 Process Water Treatment

Water will be recycled from the tailings facility to reduce the raw water consumption. One of the key risks to the high-pressure acid leach circuit availability is scaling, caused primarily by calcium. Magnesium build up in the system can also have adverse effects on slurry rheology and acid consumption. This water will be treated to remove metals and sulphates, returning it to the equivalent of raw water quality.

Process water will be treated in a high-density sludge process to remove any solids or dissolved metals, in particular magnesium and manganese. Ammonia will then be stripped from the treated water before it proceeds to an ion exchange process, which uses two circuits to remove calcium, magnesium, sulphate and other impurities. The treated process water will then be transferred to the start of the processing facility and used for dilution in ore preparation.

18.3 Power and Steam

18.3.1 From the grid

The 2005 prefeasibility study assumed the use of two natural gas fuelled gas turbine generators to provide electrical power to supplement that generated from the sulphuric acid plant's steam turbine generator. The natural gas would be supplied from a lateral pipeline connected to the existing natural gas pipeline which runs south of Condobolin. The process plant design has evolved to the point where it is more economical to replace the natural gas lateral pipeline and gas turbine generators with a grid electrical supply. The electrical supply is currently owned and operated by the New South Wales State Government power utility Essential Energy.

The Project site will tie in to the grid via the installation of a new 66 kV overhead transmission line from Trundle. This line will have a capacity of 25 MVA however, Essential Energy advise that only 7 MVA will be available for Sunrise, due to the limited capacity of the existing transmission line from Parkes.

Future power demand from Clean TeQ Sunrise will require either a new transmission line from Parkes to Trundle, or an upgrade to the existing Essential Energy transmission line (11 MVA capacity).

The new or upgraded line will provide a capacity of 30 MVA, of which 25 MVA will be available to Clean TeQ Sunrise.

18.3.2 Process Plant Power Supply

Power for the process plant will be primarily provided from the sulphuric acid plant's steam turbine generator(s) and supplemented with power from the electrical tie-line. Power for black start and emergency power conditions will be provided by the tie-line and a portable diesel-fired generator system.

18.3.3 Steam Turbine Generators

The purpose of the power station is to generate and supply electricity to the process plant. Excess high pressure superheated steam and medium pressure steam will be used to generate electricity in a steam turbine generator for the site.

High pressure steam (non-superheated but saturated) from the cogeneration plant is delivered to the two autoclaves. High pressure superheated steam for the steam turbine is supplied from the sulphuric acid plant. Medium pressure steam is also supplied from the sulphuric acid plant.

18.3.4 Auxiliary Boiler

The auxiliary boiler is a standalone package boiler that is capable of generating high pressure saturated steam. This is a conventional boiler comprising an economiser and an evaporator. It will be used primarily to start the sulphur melter and acid plant but can also be used to drive one train of PAL (using stored or imported acid) and the crystallisers, when the acid plant is off-line.

18.3.5 Black Start and Emergency Power Generation

Four diesel driven generators are provided as emergency generators. Diesel is fed to the emergency generators from the diesel storage tank.

If necessary, additional power from diesel generators at the Construction Camp can be utilised via a 11 kV power transmission line.

18.3.6 Raw Water System Power Supply

Power for raw water extraction and delivery to the process facility will be delivered using an electrical tie-line connected to the Condobolin-Forbes electrical network. This tie-line will connect to the grid at 22 kV and will require some of Essential Energy’s existing electrical network to be upgraded and extended to provide the power required for the raw water extraction and transfer.

18.3.7 Construction Camp Power Supply

The construction camp electrical power be provided by diesel engine powered generators until the electrical tie-line to the processing plant is installed and commissioned. At this point in time, a private electrical connection from the process plant to the camp will be installed and these diesel generators will provide emergency power for the camp and the processing plant in the event that the grid connection is not available.

18.3.8 Rail Siding Power Supply

Power for the rail siding will be obtained from the 415V electrical powerline which runs adjacent to the rail siding. The load is low and can be accommodated by the existing electricity network.

18.3.9 Existing Powerline

An existing 22 kV overhead electrical transmission line runs through the Project site. This powerline feeds the local communities surrounding the Project site and will be relocated to the perimeter of the site.

18.4 Tailings, water storage facilities and evaporation pond

18.4.1 Tailings Storage Facility (TSF)

All tailings generated in the process plant will be pumped to and stored in the TSF located northeast of the process plant. The TSF design has been undertaken to meet the technical requirements of the project, achieve environmental targets and to comply with applicable legislation and design codes. The TSF is assessed as having a consequence category of “High C”, in accordance with relevant Australian National Committee of Large Dams (ANCOLD) and New South Wales Dam Safety Committee (NSW DSC) guidelines, and selected design criteria reflects this ranking.

The TSF is comprised of three cells, Cell 1 to 3, with total footprint area of 380ha. Tailings will be pumped to the TSF as a slurry with a solids concentration of approximately 42% by weight. The tailings slurry will be deposited through a series of spigots located at the perimeter of the cells and a decant pond will be maintained in the corner of each cell. Withdrawal of water from the decant ponds will be managed with weir-plate controlled steel decant inlets, laid on the upstream slope of the cell embankment. The decant inlets from all three cells will be connected to a decant pipe buried below the embankments, which will discharge into a solution sump. The decanted water will be pumped from the sump to the Water Storage Dam, for eventual reuse in the process plant.

The three cells will be filled one cell at a time and raised progressively to a maximum elevation of 314 mRL AHD. The cell walls will be raised using downstream construction methods so that the containing embankment is founded wholly on natural soils. Experience with nickel tailings in Australia and overseas has shown that upstream raising is not possible due to the low strength of the placed tailings and their potential to liquefy. A spillway will be incorporated in the southern embankment of the first cell to be built (Cell 2) at its ultimate height to provide emergency discharge for the TSF.

18.4.2 Water Storage Dam

The Water Storage Dam will be used to store the decant water from the TSF and rainfall runoff from within the local catchment. It will be located in a small valley immediately north of the TSF, which minimises the earthworks required to create capacity. The dam is assessed as having a consequence category of “Significant”, in accordance with relevant ANCOLD and NSW DSC guidelines, and selected design criteria reflects this ranking.

The Water Storage Dam will cover an area of 58ha and be lined with 1.5 mm high density polyethylene. It has been designed with a spillway sized to pass the critical duration 1:10 000 Annual Exceedance Probability (AEP) flood, in accordance with the ANCOLD Guidelines.

18.4.3 Evaporation Pond

A 2.6 ha single cell evaporation pond will manage a minor stream of high chloride process water. To prevent chloride build-up in process water, this outflow from the process plant is separated from the TSF system and retained and evaporated in an evaporation pond. The evaporation pond design incorporates compacted earth embankments (max. 3m high) with a low permeability central cut-off and core. The evaporation pond has been sized to contain all water from the waste inflow and rainfall during a 20-year simulation using the cumulative wettest sequential 20 years of rainfall data from the scientific information for land owners (SILO) rainfall data record.

18.5 Water Management

18.5.1 Water Treatment Plant Sludge Pond

The Water Treatment Plant Sludge Pond (Sludge Pond) will be used to store the waste stream from the water treatment plant generated over a 12-month period and incidental rainfall. The rate of inflow is anticipated to be 122,690 m^3/year. The initial Sludge Pond is located northeast of the TSF, adjacent to the Evaporation Pond, outside the extent of the 1:100 AEP flood event. Additional Sludge Pond cells will be constructed to the southeast of the initial Sludge Pond, following construction of the Southern Diversion drain.

18.5.2 Clean Water Diversion Design

The mine site is crossed by ephemeral drainage valleys running southwest to northeast (Figure 18-4), that includes:

- northern valley, which divides the site into its northern and southern branches. This valley will be Cut-Off by the mine pits sometime after Year 5, requiring significant diversion works (the northern diversion)
- central valley, which is confluent with the northern valley within the Fifield State Forest. This valley will be Cut-Off by mine pits some time before Year 5, requiring diversion works into the northern valley prior to the natural confluence point
- southern valley, which flows through the proposed TSF, requiring less extensive diversion works (the southern diversion).

Various other catchments within the site will also require diversion works, as will drainage and diversion works for mine facilities including mine pits, mine plant, TSF, Evaporation Ponds and Sludge Pond. Drainage works will be implemented in a staged approach as required over the life of mine.

18.5.3 Storm Water Management

Stormwater management and stormwater infrastructure design is based on the Australasian and International guidelines (Managing Urban Stormwater: Soils and Construction, Landcom, 2006 and the International Erosion Control Association, 2008). The primary objective was to control mine-impacted water.

The overall stormwater management strategy includes:

- appropriate management of the movement and separation of water throughout the site
- drainage control measures aimed at preventing or reducing soil erosion caused by concentrated flows
- erosion control measures aimed at preventing or reducing soil erosion caused by wind and/or rain drop impact and sheet flow (i.e. the control of splash and sheet erosion)
- sediment control measures aimed at trapping and retaining suspended sediments
- maintaining the stability, integrity and effectiveness of embankments, adjacent landforms, and associated stormwater management systems and structures.

As a general principle, undisturbed stormwater runoff will be diverted around disturbed areas of the site. Water collected in mining pits and runoff from waste dumps will be temporarily contained in sediment basins and recycled, evaporated or assessed to meet surface water discharge requirements prior to discharge to the environment. Sediment basins were sized to detain sufficient volume of runoff from disturbed areas for settling and storage of sediment.

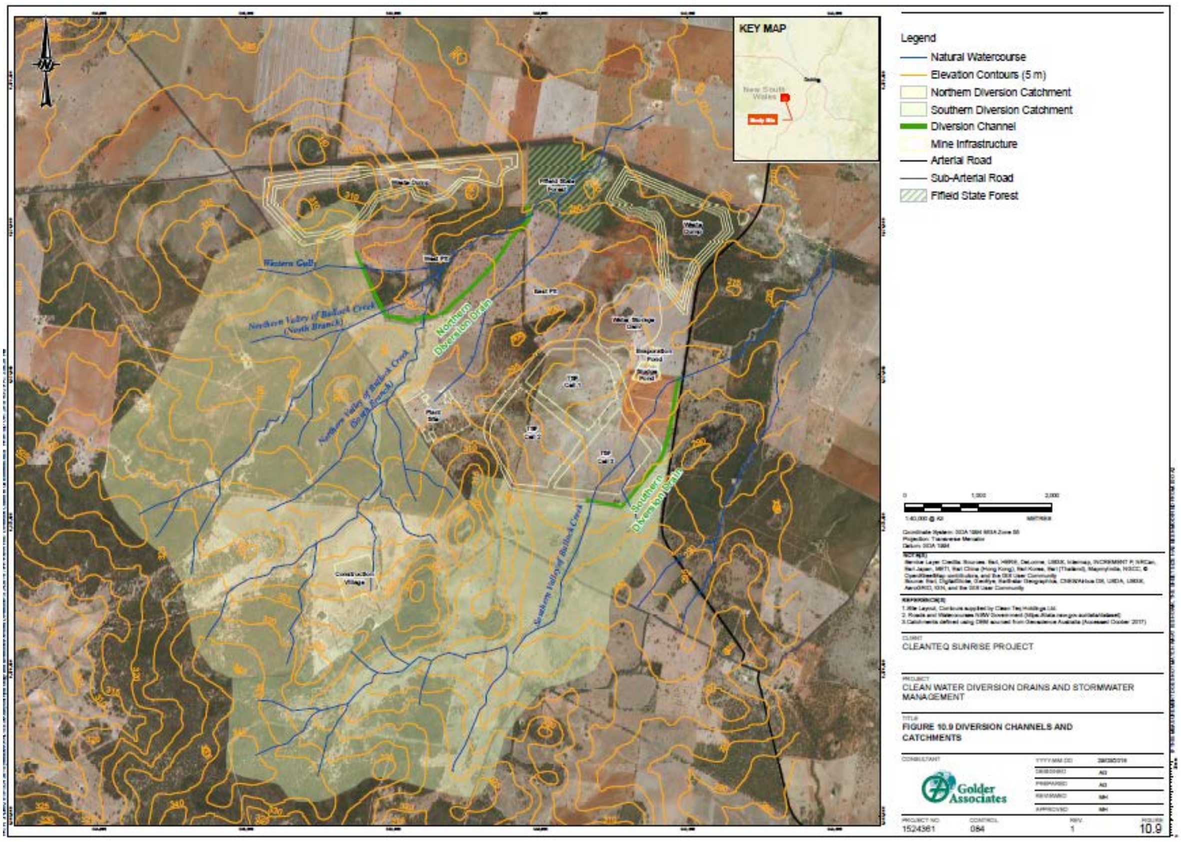

Figure 18-4: Diversion Channel Catchments

18.5.4 Decommissioning and Rehabilitation

The preliminary design of Clean TeQ Sunrise TSF, Sludge Dam, Evaporation Pond and water management infrastructure has addressed the environmental considerations that need to be satisfied throughout the operational life of these facilities. A closure plan for the facilities will be developed in consideration of the design and the guiding principles and commitments made by Clean TeQ.

The following key items will be addressed in the detailed design for closure:

- groundwater seepage - reduce long-term release of contaminants to the environment
- dust management - reduce the release of fine particles
- surface erosion - avoid the development of severe gullies that may compromise tailings containment, sediment management and the overall aesthetics
- sediment management - reduce the long-term release of solids to the external environment.

18.5.4.1 TSF Closure

The important conceptual aspects of the decommissioning of the TSF include placement of granular cover material over final tailings surface temporarily to store infiltrated water arising from incident rainfall, reshaping of surfaces to allow for water shedding, and then adding topsoil and vegetation to all surfaces. Detailed planning and design of the rehabilitation measures for the TSF will take place during operation of the facility, well ahead of planned closure.

18.6 Fuel Storage and Transfer

Diesel will be the only fuel used on site and will be stored and distributed in two locations - adjacent to mining operations and within the processing plant.

18.6.1 Mining Operations

The mining fuel and lubricant facilities will include the following and will consist of temporary and permanent facilities. They will be managed by the mining contractor.

- bunded bulk fuel (diesel) storage with volume capacity for at least ten days of mining operations
- lubricants and hydraulic oils stored in self-bunded containers, intermediate bulk containers (IBC) or drums. Both full and empty IBCs and drums will be stored in fully bunded areas
- site service truck with appropriate storage capacities for diesel, lubricants and hydraulic oil
- heavy vehicle refuelling facility with appropriate spill management infrastructure
- light vehicle refuelling facility with appropriate spill management infrastructure.

The estimated volume requirements will be subject to the actual supply logistics for Clean TeQ Sunrise.

18.6.1.1 Vehicle Washdown Facility

The vehicle washdown facility will be used for cleaning of all mobile mining equipment and light vehicles. The washdown facility will also be used for washing some machine components. Sludge will be removed from a collection sump by front end loader and an oil/water separator will capture grease and oil for disposal. The vehicle washdown facility will be located in close proximity to the permanent maintenance facilities to enable the washing of equipment prior to planned maintenance.

18.6.1.2 Hydrocarbon Management

Hydrocarbons will be managed within the maintenance workshop. Waste oils and grease will be collected and pumped to bulk storage tanks by oil evacuation pumps. The permanent workshops will provide for this with additional bunded areas for storage of drums of oils, greases and coolants.

18.6.2 Processing Plant

The processing plant diesel storage and distribution facilities provide diesel for plant vehicles and to service intermittent requirements including fuel for generator sets, the fire water pump, the boiler for sulphur melting furnace pre-heat.

The diesel storage and transfer facilities include a 90kL self-bunded diesel storage tank including transfer pumps, a 10 kL self-bunded diesel storage tank fitted with high and low-flow fuel bowsers.

In addition, facilities to install temporary self-bunded tankage to a volume of 200 kL will be provided for periods of high diesel demand during start-up, and diesel tanker unloading facilities will be provided at both the 90 kL and 200 kL diesel storage facilities.

18.7 Roads

18.7.1 External Road Upgrades

Upgrades to the existing road network to accommodate the proposed mine site location are planned within the local Shires. The scope of work has been defined based upon NSW Department of Planning and Environment requirements which include preparation and submission of a road upgrade and maintenance strategy.

The extent of road upgrades is shown in Table 18-1, with the proposed intersection upgrades shown in Table 18-2. The location of the road upgrades is shown in Figure 18-5.

Table 18-1: Extent of Road Upgrades

Bogan Way to MPF Access Road			Road Upgrades Required
Location	**Existing Condition**	**Estimated Length (km)**	
Fifield-Trundle Road - between The Bogan Way and Platina Road (SR171)	Sealed, 6 m wide typical	8.7	8.0m seal with 1.0 m unsealed shoulders each side, 3.0 m wide gravel shoulders for 30 m each side of minor roads and property accesses to be sealed.
Platina Road -between Fifield - Trundle road and Fifield road (SR64)	Sealed, 3.5 m wide typical	10.1	8.0m seal with 1.0 m unsealed shoulders each side, 3.0 m wide gravel shoulders for 30 m each side of minor roads and property accesses to be sealed.
Fifield road - between Platina Road and Slee Street (MR57N)	Sealed 6.5 m wide - 2 km of 5 m road approaching Fifield	5.2	Seal should be widened to 9.0 m with 1.0 m unsealed shoulder.
Slee Street -between Fifield Road and Wilmatha Road	Sealed, 20 m wide formation 16 m wide from gutter to gutter	0.4	Not required.
Wilmatha - between Slee Street and MPF site access Road (SR34)	Gravel road 6m wide	5.2	8.0 m seal with 1.0 m unsealed shoulders each side, 3.0 m wide gravel shoulders for 30 m each side of minor roads and property accesses to be sealed.
Bogan Way			
The Bogan Way between Henry Parkes Way and Fifield-Trundle Road (MR350)	Sealed, various widths	30.3	Maintenance only.

Bogan Way to MPF Access Road			Road Upgrades Required
Location	**Existing Condition**	**Estimated Length (km)**	
The Bogan Way between Trundle and Fifield-Trundle Road	Sealed, various widths	TBA	Section of The Bogan Way between Trundle and Fifield-Trundle Road may need to be widened to a 9.0 m seal with 1.0 m unsealed shoulders subject to the level of background traffic growth outside of Trundle.
Middle Trundle Road			
Middle Trundle Road (SR 83)	16.8 km from The Bogan Way is gravel	29.5	Maintenance only. No specific upgrade required.
-	Remainder is Bitumen sealed road	-	Note the appropriate standard for Middle Trundle Road if sealed is a 7.2 m wide sealed carriageway with 1.0 m wide unsealed shoulders each side.

Table 18-2: Intersection Upgrades

Location	Required Upgrade
The Bogan Way and Henry Parkes Way	Line marking and signage to be improved as part of ongoing maintenance.
The Bogan Way and Fifield -Trundle Road	Four-way intersection design to incorporate right-left staggered layout of Scotson Lane and Fifield-Trundle Road and to minimise stacking and queuing risks at the adjacent railway level crossing.
Fifield-Trundle Road and Limestone Quarry Access Road	BAL/BAR treatment in Fifield-Trundle Road.
Platina Road and Fifield Road	Line marking and signage to be improved as part of ongoing maintenance.
Slee Street and Fifield Road	No specific upgrade, subject to maintenance.
Slee Street and Wilmatha Road	Advance warning signs on: Slee Street (W9-1R) Fifield Road (W9-2L) Wilmatha Road (W3-2).
The Bogan Way and Middle Trundle Road	Line marking and signage to be improved as part of ongoing maintenance. Give way (R1-2) and advance warning (W3-2) signs on Middle Trundle Road.
Henry Parkes Way and Middle Trundle Road	Line marking and signage to be improved as part of ongoing maintenance.
Henry Parkes Way and Fifield Road	Line marking and signage to be improved as part of ongoing maintenance. Give way (R1-2) signs on Fifield Road.
Springvale Road and Fifield Road	Line marking and signage to be improved as part of ongoing maintenance.

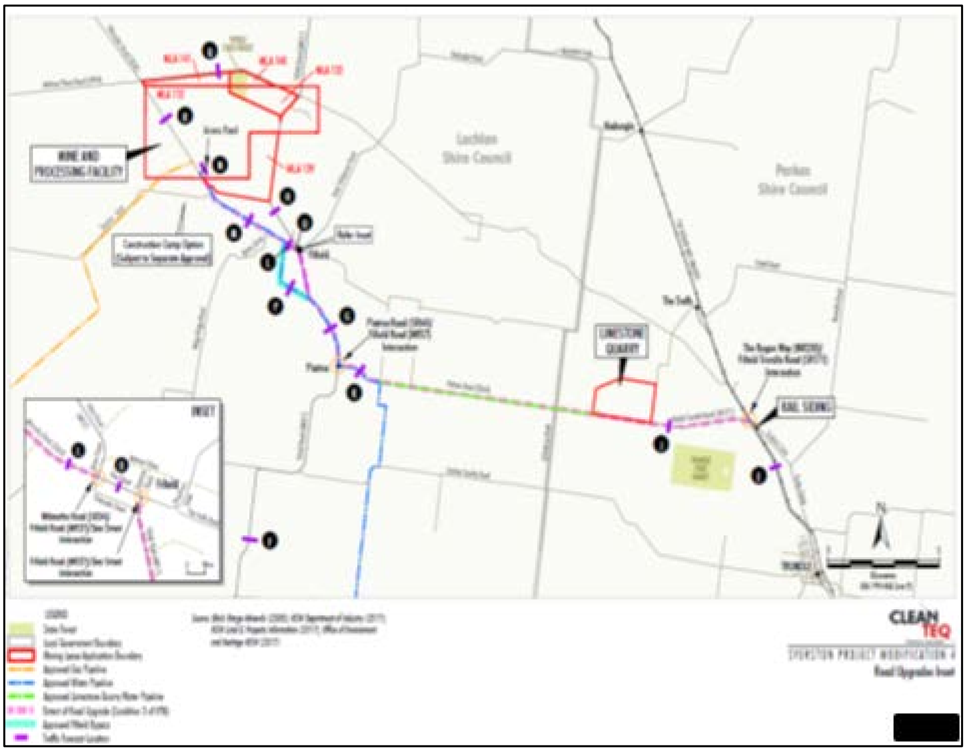

Figure 18-5: Location of Road Upgrades

18.8 Construction Camp

The Clean TeQ Sunrise Camp facilities will be located on a property adjacent to the mining leases and to the southwest of the process plant. They are located 2.5 km from the process plant.

The Camp will be constructed in stages, with the first stage installing key infrastructure and amenities (dining, recreational facilities etc.) and rooms sufficient to accommodate 300 people. After the first stage installation of 300 rooms, it is planned to add 100 rooms per month up to a final Camp capacity of 1,300 persons. This strategy allows for early access to the Camp and the orderly expansion of the following stages, while maintaining separation of the camp occupants from the remaining camp construction.

Clean TeQ have purchased all buildings within the first stage of the camp and will retain this facility following commissioning, for use as shutdown accommodation during operation. All other buildings will be demobilised.

The Camp will include sleeping quarters, laundry and ablution facilities, kitchen/dining complex, recreation facilities, camp management and maintenance facilities, first aid facility and a retail outlet. It will be serviced by water and power supply facilities (initially diesel generators, being replaced by grid power as it becomes available), a packaged sewerage system, waste water irrigation area and a data and telephone system connected to the national fibre and wireless networks. The layout of the camp will take advantage of the north facing slope for surface water drainage and sewage transfer.

Buildings will be pre-fabricated, modular structures with split system air-conditioning to appropriate areas. The buildings supply contractor shall design, supply and construct all components of the camp other than the transportable buildings.

The Camp will be run by a suitably qualified external contractor and will operate on a motel style basis whereby rooms will only be assigned to residents for the duration of their rostered cycle. When residents leave for their rostered time off their personal belongings will be stored in a secure facility. Upon return, the resident will be assigned a new room.

A limited number of senior staff quarters will be designated for camp management and supervisory staff.

Catering will be provided by a dry mess facility incorporating a commercial kitchen for food preparation and dining areas for the consumption of sit-down meals. The facility will also provide for the preparation of crib meals by residents for consumption at the work site.

19 Market Studies and Contracts

The Sunrise Project will be a major global supplier of three high-purity products to rapidly growing markets and will produce these in the lowest quartile of the industry cost curve. High-purity nickel and cobalt sulphate products will target cathode manufacturers supplying the rapidly growing lithium-ion battery market.

The global market for scandium as a specialist alloying metal for high-strength, lightweight materials has enormous potential growth, as new deposits like Clean TeQ Sunrise are developed.

Offtake commitment for annual tonnages representing approximately 20% of the Project's nickel and cobalt production for the first five years has already been secured, with a target of 80% of production to be committed under binding offtake agreement by final investment decision.

19.1.1 Clean TeQ Sunrise Products

The Project will produce the following products (on an annual basis averaged over the life of mine). The annual production quantity of each product will vary based on the mine planning optimisation:

- 82,000 tpa of nickel sulphate – primarily for the lithium-ion battery market
- 16,100 tpa of cobalt sulphate – primarily for the lithium-ion battery market
- 10 tpa of scandium oxide – market developing but essentially a high value alloying material.
- 82,000 tpa ammonium sulphate as a by-product – used primarily as an agricultural fertiliser.

19.1.2 Nickel

19.1.2.1 Nickel Supply and Demand Forecast

Growth in the lithium battery market is being primarily driven by demand for electric vehicles. A strong correlation between renewables generation capacity and installed storage capacity has also been observed at a global level since 2010. For lithium-ion battery demand, total energy capacity is more important than power capacity as this will inherently determine the nickel and cobalt content. Global annual stationary storage installations are expected to grow from less than 1 GWh in 2015 to 65.6GWh in 2025, adding significant upside to battery demand from electric vehicle growth forecasts. Refer Figure 19-1 and Figure 19-2.

Demand for nickel sulphate primarily for battery use is forecast to grow at an annual rate of 22.5% from 47,000t (in nickel units) in 2016 to 293,000t (nickel units) in 2025. The current forecast nickel sulphate supply response is unlikely to satisfy that demand. As a result, the market for nickel sulphate is likely to be in supply deficit by 2022 (see Table 19-1 and Figure 19-3).

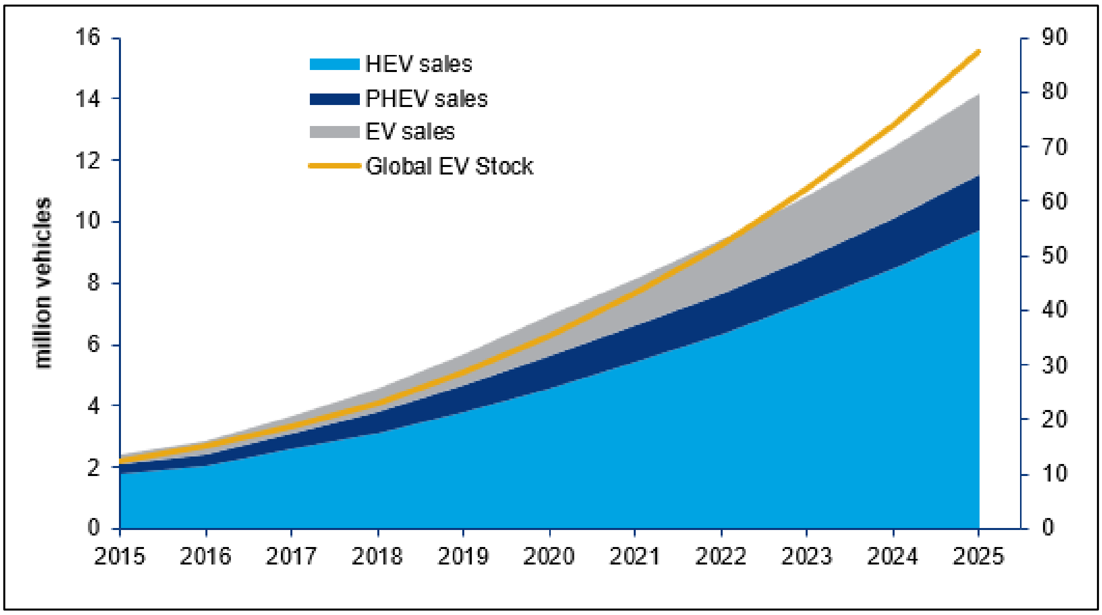

Figure 19-1: Electric Vehicle Sales by Type[24]

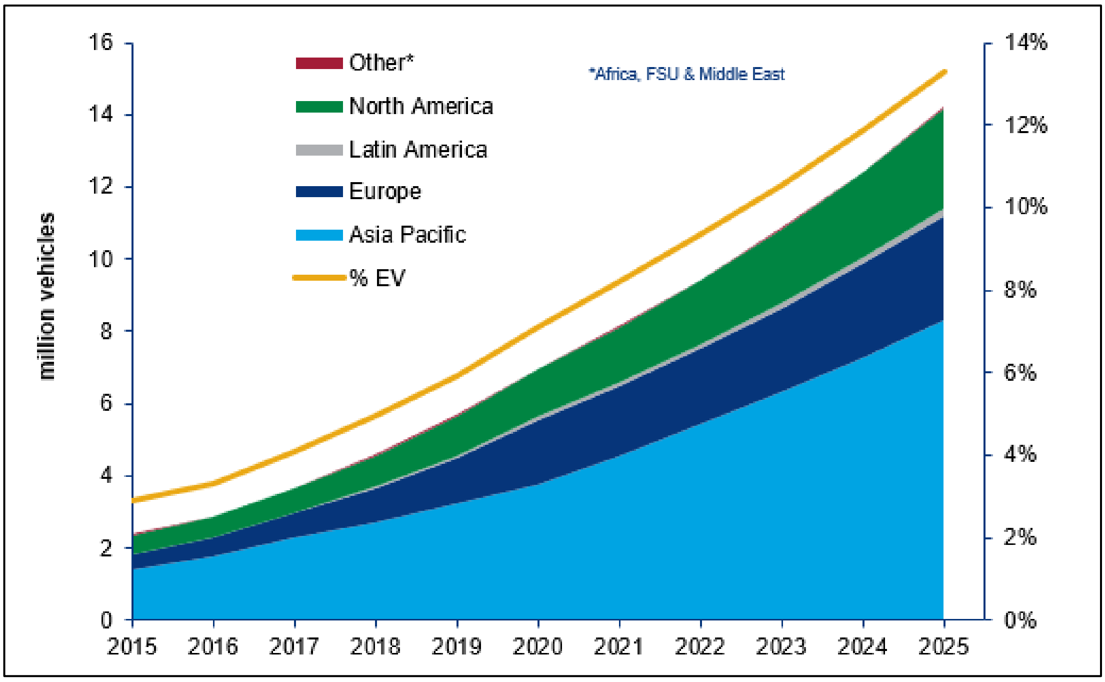

Figure 19-2: Electric Vehicle Sales by Geography[25]

Table 19-1: Nickel Sulphate ($NiSO_4.6H_2O$) Demand Estimate and Forecast (kt Cont Ni)

Demand Area	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Plating	12	13	13	13	13	14	14	14	15	15
Catalysts	6	6	6	6	6	6	6	6	6	6
Batteries	47	60	78	100	132	154	179	210	247	293
Total	65	79	96	119	151	173	199	230	267	314

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[24] Source: Wood Mackenzie. (EV – Electric Vehicle, HEV – Hybrid Electric Vehicle, PHEV – Plug-in Hybrid Electric Vehicle).
[25] Source: Wood Mackenzie.

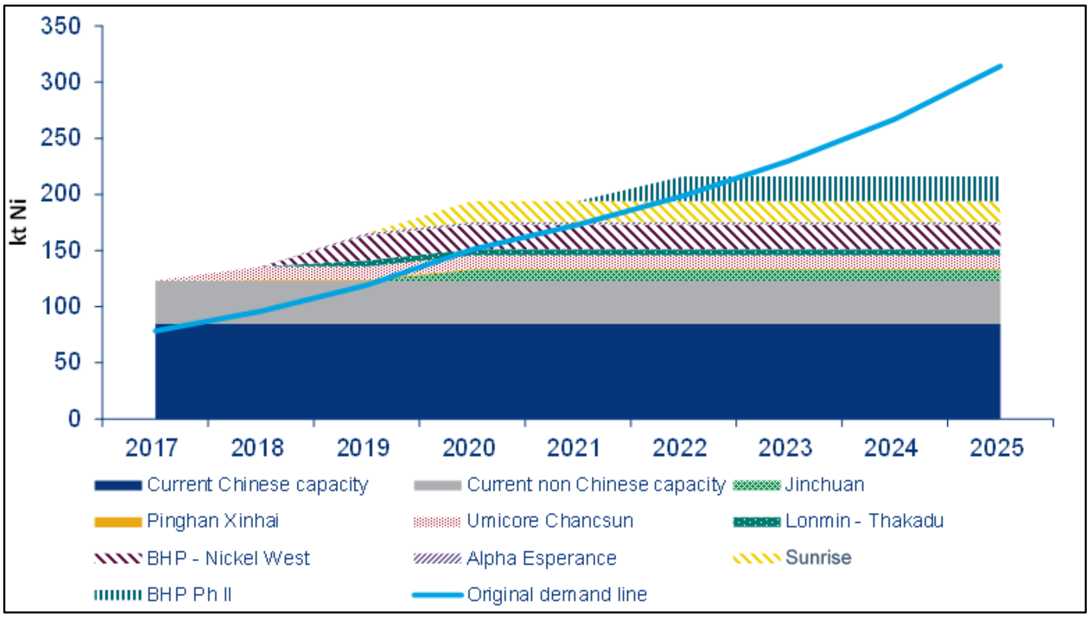

Figure 19-3: Demand vs Capacity in Nickel Sulphate (kt Contained Nickel)[26]

19.1.2.2 Nickel and Nickel Sulphate Price Forecast

In its economic analysis for the DFS, Clean TeQ has assumed a flat London Metal Exchange nickel price of USD15,430/t (USD7/lb) on a real basis for the life of the Project (see Figure 19-4). Continued development of the electric vehicle industry is, in the long-term, likely to support higher nickel prices.

In recent years, the nickel pig iron industry emerged as a source of supply and the marginal cost of production of nickel pig iron effectively represented a cap on the nickel price. Whilst nickel pig iron is suitable for stainless steel applications, lithium-ion batteries typically require Class 1 nickel products and as such, nickel pig iron is not currently suitable for use in these batteries. Furthermore, conversion of nickel in nickel pig iron to high purity nickel sulphate is currently prohibitively expensive such that it is not a viable route compared with developing new high-purity nickel production. Consequently, in the past twelve months the nickel price has risen significantly in response to demand from the electric vehicle sector.

Further, Wood Mackenzie's analysis has concluded that the long-term incentive price for development of new nickel projects is USD9/lb.

During the last twelve months, the price premium for battery grade nickel sulphate has been USD3,687/t on average. In its economic analysis for the DFS, Clean TeQ has assumed that battery grade nickel sulphate produced at the Project will sell for a premium of USD2,205/t (USD1/lb) (see Figure 19-5) more than the London Metal Exchange nickel price.

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[26] Source: Wood Mackenzie and Beijing General Research Institute of Mining and Metallurgy (BGRIMM)

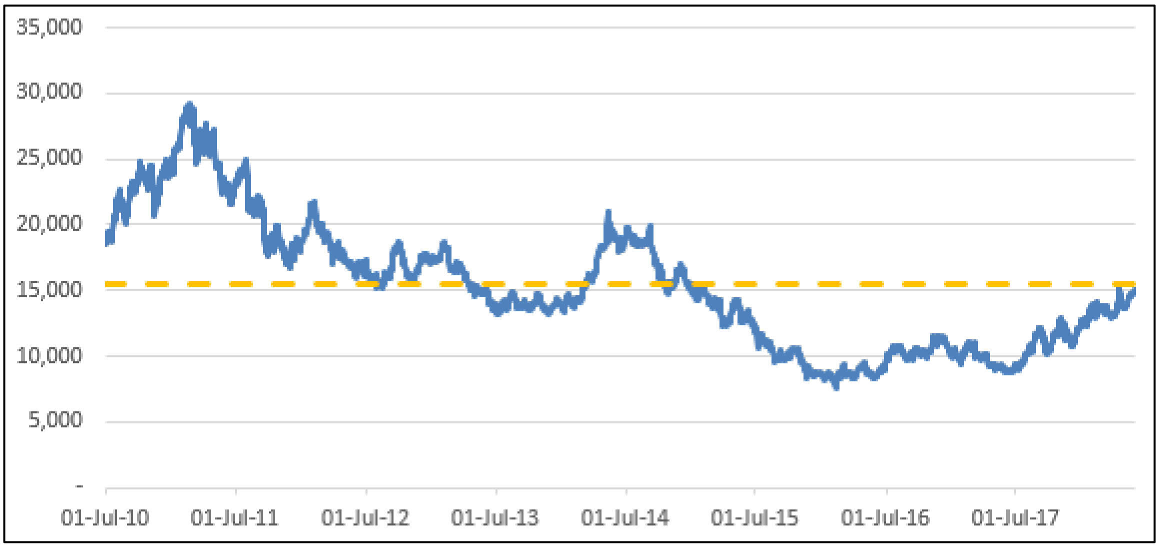

Figure 19-4: LME Historical Nickel Price Since 2010

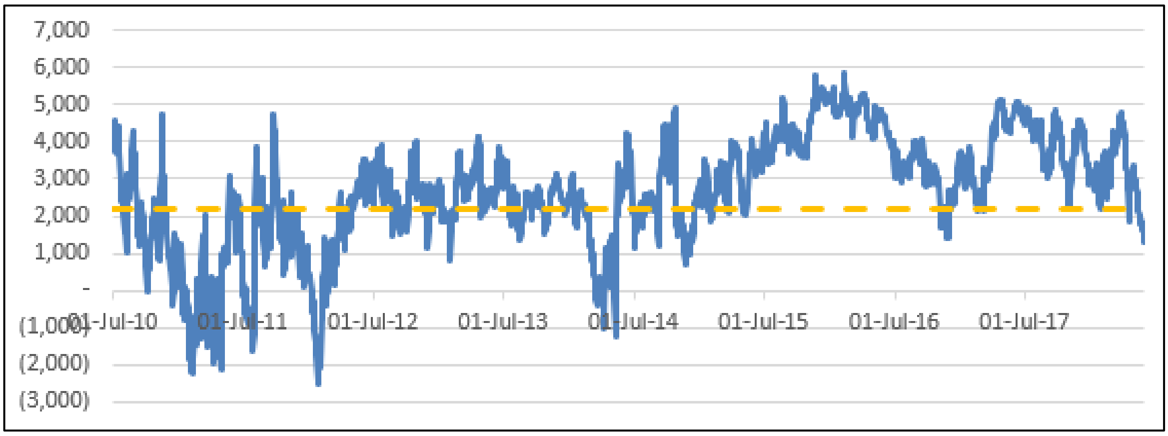

Figure 19-5: LME Nickel vs Chinese Battery Grade $NiSO_4$ Price (NiEq Basis)[27]

19.1.3 Cobalt

19.1.3.1 Cobalt Supply and Demand Forecast

Cobalt is used in lithium-ion battery precursor production as a sulphate or as an oxide. In addition, cobalt metal powder and cobalt hydroxide are used to produce rechargeable alkaline batteries.

Cobalt demand for use in batteries is forecast to grow at an annual rate of 15.6% from 52,500 t in 2016 to 215,000 t by 2025. Total global cobalt demand is forecast to increase from 99,000t in 2015 to 274,000 t by 2025.

Cobalt supply is constrained by several factors. It is typically produced as a by-product at nickel and copper mining operations, so is inextricably linked to demand for nickel and copper and supply is less sensitive to cobalt prices. Over 60% of global cobalt production is sourced from the Democratic Republic of Congo. Media and NGO reports have documented accusations of human rights abuses and child labour at artisanal cobalt mines in the Democratic Republic of Congo. Calls for increasing supply chain transparency from NGO's is pushing major end users of lithium-ion batteries including Apple Inc., Tesla Inc., General Motors Company and Volkswagen AG, to focus on the auditability of the cobalt supply chain. The Project is expected to be a globally significant producer of cobalt from a

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[27] Source: Antaike, Bloomberg data stream, Excludes VAT in China.

western democracy which is renowned for its political stability and high standards of health, safety, environmental and community practices.

Independent market consultant Wood Mackenzie's forecast of cobalt supply is shown in Figure 19-6 below with production from identifiable sources increasing from 97,000 t in 2016 to 130,000 t in 2025 based on production from operating mines and foreseeable projects.

This leads to a significantly large demand/supply gap. With copper mines in the Democratic Republic of Congo and Zambia contributing 74% of the mined production of cobalt, to meet this demand/supply gap will require more than doubling output from the African Copper Belt. While this is not infeasible, given that the copper market itself is expected to be in short supply by 2025, it is more likely that prolonged cobalt shortage or its anticipation will lead producers to experiment with lower cobalt content chemistries in their batteries which will eventually bring the market back into balance.

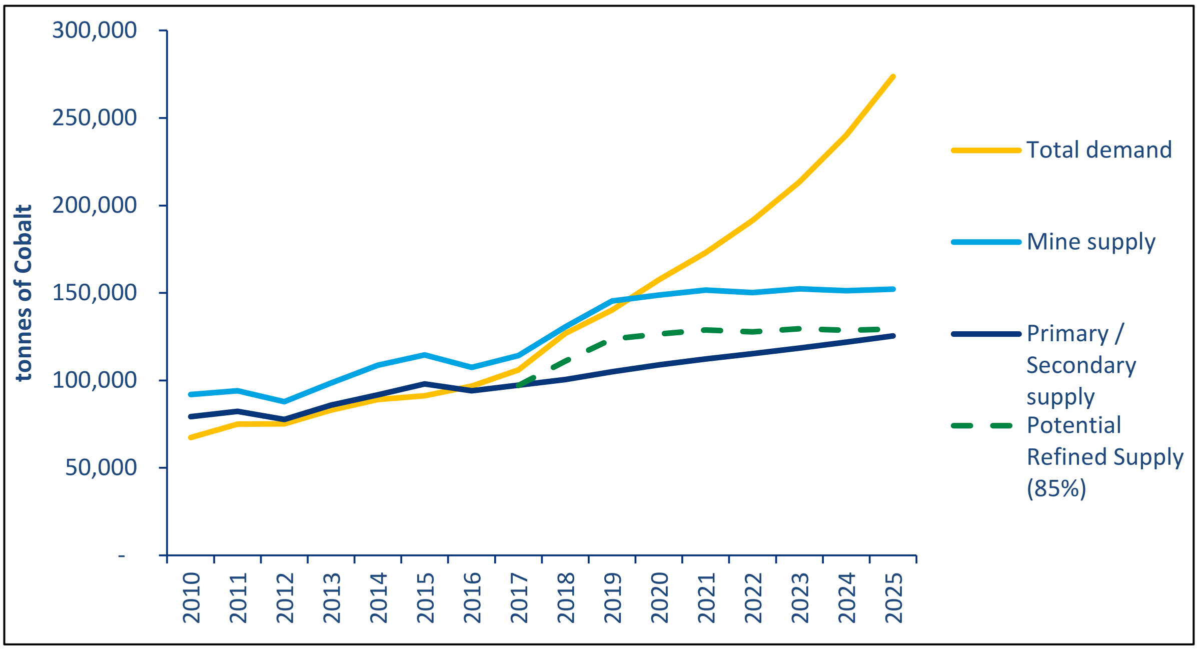

Figure 19-6: Forecast of Cobalt Supply vs Demand

19.1.3.2 Cobalt and Cobalt Sulphate Price

In its economic analysis for the DFS, Clean TeQ has assumed a flat London Metal Exchange/London Metal Bulletin cobalt price of USD66,140/t (USD30/lb) on a real basis for the life of the mine. At the time of preparing the Definitive Feasibility Study the spot cobalt price was 33% above these levels after increasing significantly in the past twelve months in response to growing demand from the electric vehicle sector.

The cobalt price assumption adopted by Clean TeQ is broadly in line with long term consensus bank/ broker forecasts.

Wood Mackenzie's view is that the long-term price of cobalt will be USD20/lb, although its analysis concludes that a reduction in prices will require an as yet unforeseen supply response. If there is a delay in the supply side response then Wood Mackenzie forecasts cobalt prices to be USD38/lb by 2025, whilst still maintaining its eventual long-term view on prices.

Deviations both above and below the London Metal Exchange price occur because of short term demand and supply imbalances. For the DFS, Clean TeQ has assumed that the cobalt sulphate price (metal equivalent basis) will be equal to the London Metal Exchange price of cobalt.

19.1.4 Scandium

The market for scandium has very high latent potential. Historically, low volume sales for niche applications have been constrained by lack of global supply (volume and diversity) and high prices. Recently, five of the world's highest grade deposits of scandium have been discovered in Australia, including Clean TeQ Sunrise, and together they have the potential to transform the market for lightweight alloys. Large, long-term supply has the potential to trigger widespread adoption of the material for a range of applications in the aerospace, automotive, rail and marine industries.

Historically, scandium oxide prices range from USD2,000-4,000/kg, while forward prices of USD1,500/kg have been assumed for the Project, which is significantly lower than prices quoted in modelling of other scandium projects, however, based on feedback from potential customers and internal Company analysis, Clean TeQ believes this price is required to drive significant demand for scandium aluminium alloys for many of the high value markets it is targeting.

19.1.5 Ammonium Sulphate as a By-Product

The Project will also produce approximately 82,000tpa of ammonium sulphate as a by-product. This will be packaged and sold mainly as an agricultural fertilizer. The assumed sales price for ammonium sulphate established for the DFS was USD90/t free on board (FOB)[28].

19.1.6 Marketing Program

Clean TeQ has entered into a binding take-or-pay offtake agreement with Beijing Easpring for annual tonnages representing approximately 20% of the nickel sulphate and cobalt sulphate production from the Project for the first five years. Clean TeQ is currently progressing discussions with other potential customers on substantially similar terms and is targeting having offtake agreements in place for approximately 80% of production prior to a final investment decision.

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[28] Incoterms 2010.

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.

Figure 20-1 and Figure 20-2 show Clean TeQ Sunrise's landholding over the Project area and the Westella limestone deposit. These properties are Syerston, Sunrise, Kingsdale, Slapdown and Westella. Agreements to acquire or option The Troffs and rail siding properties are currently being negotiated.

The main project area is covered by two adjoining tenements, Mining Lease (ML) 1770 and a portion of Mining Lease Application (MLA) 113. These two tenements are underlain by Exploration Licence (EL) 4573. ML 1769 covers the Westella Limestone deposit which is underlain by Exploration Licence EL 8561. All of the MLs, MLAs and ELs covering the main project area are 100% controlled by Clean TeQ via the 100% owned subsidiary, Clean TeQ Sunrise Pty Ltd, as well as freehold ownership of the majority of the project area and water rights for the Project. Land currently not owned by the company which is required for the project is under negotiation for purchase or an access agreement

The company has a water licence for 3.2 GLpa. from a bore field located approximately 70 km south of the Project. A water pipeline will be constructed to supply water to the project and has been allowed for in the capital estimate. The borefield and water pipeline were a part of the EIS completed on the Project.

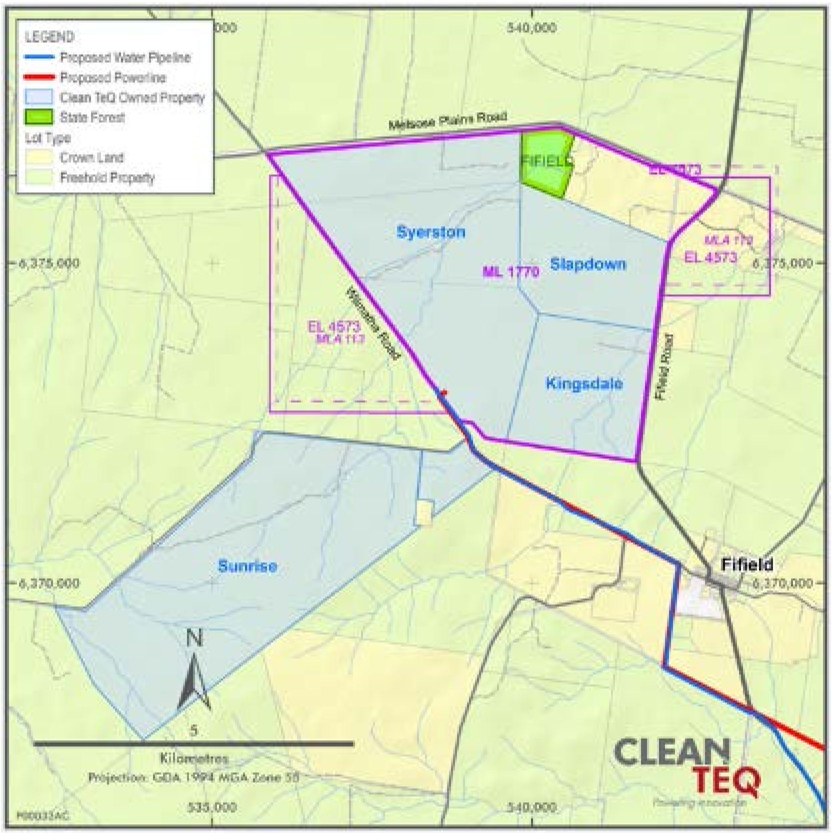

Figure 20-1: Clean TeQ Sunrise Pty Ltd Landholding - Main Project Area

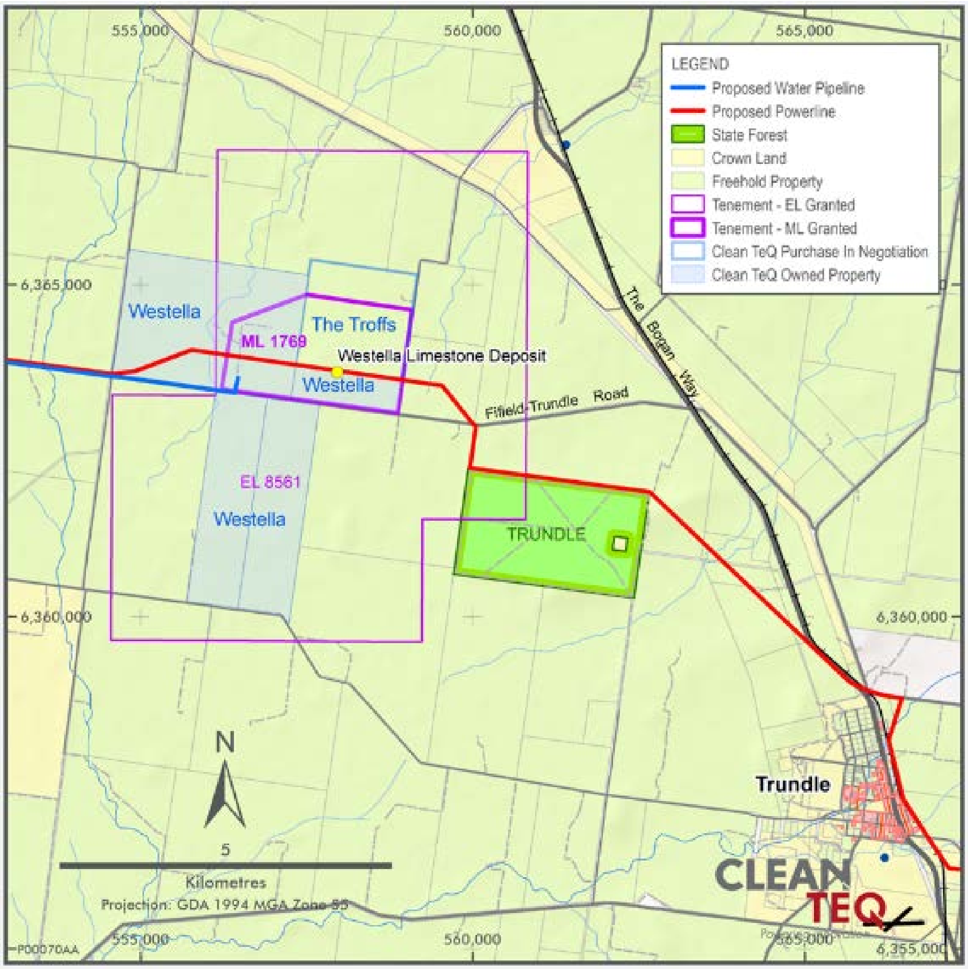

Figure 20-2: Clean TeQ Sunrise Pty Ltd Landholding - Limestone Quarry

20.1.1 Native Title and freehold land

The 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 Clean TeQ Sunrise), and thus there is minimal risk or any future project exposure to Native Title claims.

Land acquisition and option agreements and easements are proposed to secure 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 which is the subject of the rail siding (documented and binding though not yet completed/settled).

20.2 Statutory Approvals

Clean TeQ is required to obtain a number of environmental approvals, licences and permits under applicable legislation in order to lawfully develop and operate the Project.

The key legislation (that is, legislation which has a whole of project effect) is:

- NSW Environmental Planning and Assessment Act 1979 which relates to the required Development Consent for Clean TeQ Sunrise
- NSW Protection of the Environment Operations Act 1997 which relates to the required Environment Protection Licence to manage the potential environmental impacts of Clean TeQ Sunrise

- NSW Mining Act 1992 which relates to the required mining leases for the Clean TeQ Sunrise before mining operations can commence
- NSW Water Management Act 2000 which relates to the required water access licences and works approvals to meet the water demand of the Project.

Clean TeQ must also meet the requirements of other applicable legislation (for example, Biodiversity Conservation Act 2016 (BC Act) and NSW Crown Lands Act 1989[29]) which relate to various components of Clean TeQ Sunrise.

Section 20.2.1 summarises the environmental approvals, licences and permits that are or may be required under the key legislation and other applicable legislation.

20.2.1 Summary of Required Environmental Approvals, Licences and Permits

Table 20-1 summarises the environmental approvals, licences and permits for the Project that are or may be required and their status as at 30 June 2018.

As shown in Table 20-1, Clean TeQ has made significant progress in obtaining the required environmental approvals, licences and permits for the Project. In particular:

- Development Consent (including MOD 3, MOD 4, MOD 5 and MOD 6) under NSW Environmental Planning and Assessment Act 1979 is in place
- Mining Leases under NSW Mining Act 1992 for the main project area and the limestone quarry have been granted
- Water Access Licences and associated works approvals under NSW Water Management Act 2000 have been issued.

20.2.2 New South Wales Environmental Planning and Assessment Act 1979

Development consent for Clean TeQ Sunrise is held by Clean TeQ's wholly owned subsidiary, Clean TeQ Sunrise Pty Ltd. The Project has progressed through several Modifications since the original Development Consent. Modification 3 and subsequent Modification applications have occurred under Clean TeQ's ownership of the Project.

20.2.2.1 Development Consent DA 374-11-00

Development Consent (DA 374-11-00) for Clean TeQ Sunrise was issued to Black Range Minerals Ltd (previous Project owner) on 23 May 2001 by the Minister for Urban Affairs and Planning under Sections 76(A)9 and 80 of the NSW Environmental Planning and Assessment Act 1979 (EP&A Act).

20.2.2.2 Modification 1

Modification 1 (MOD 1) of the Development Consent was approved by the then NSW Department of Urban Affairs and Planning on 15 October 2005. MOD 1 authorised an increase in the autoclave feed rate to 2.5 Mtpa, limestone quarry extraction rate to 790,000tpa and adjustments to ore processing operations.

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[29] NSW Crown Lands Act 1989 will be repealed upon commencement of relevant provisions of the Crown Land Management Act 2016, expected in 2018.

20.2.2.3 Modification 2

Modification 2 (MOD 2) of the Development Consent was approved by the then NSW Department of Urban Affairs and Planning on 6 January 2006. MOD 2 authorised a reconfiguration of the water supply borefield.

20.2.2.4 Modification 3

Modification 3 (MOD 3) of the Development Consent was approved by the authorised delegate of the NSW Minister for Planning on 12 May 2017. The MOD 3 approval, among other changes, authorised the production of scandium oxide in addition to the already approved production of nickel and cobalt.

MOD 3 involves a small scale initial scandium oxide focused production phase (Initial Production Phase) prior to full scale production including scandium oxide and nickel and cobalt precipitate (Full Production Phase). The Initial Production Phase will include:

- preferentially mining scandium rich areas of the Clean TeQ Sunrise deposit, within the approved mine and processing facility surface development area, at a run of mine ore production rate of 100,000tpa
- minor adjustments to processing operations to allow for the production of approximately 80tpa of scandium oxide and up to 1,000tpa of nickel and cobalt metal equivalents as either sulphide or sulphate precipitate products.

Under the MOD 3 approval, the Tailings Storage Facility and evaporative system during the Initial Production Phase would be smaller than the approved Tailings Storage Facility and evaporative system due to the lower production rate. The modified evaporative system during the Initial Production Phase would provide appropriate storage capacity for the Tailings Storage Facility rainfall event design criteria (i.e. 1 in 100-year annual recurrence interval). The Tailings Storage Facility and evaporative system would be developed to the approved size during the Full Production Phase.

The MOD 3 approval was a significant project milestone because it allows Clean TeQ to produce the valuable commodity, scandium oxide and gives Clean TeQ the flexibility to initially develop a revenue generating smaller-scale project pending the full-scale project.

20.2.2.5 Modification 4

Notwithstanding the favourable outcomes of the MOD 3 approval, Clean TeQ has identified further opportunities to realise additional project efficiencies. On 4 May 2017 Clean TeQ applied to the NSW Department of Planning and Environment for a Modification 4 (the MOD 4 application) of the Development Consent. Modification 4 application followed Clean TeQ’s Project Optimisation Reports of March 2017 which identified potential opportunities to:

- increase the efficiency of mining and processing operations
- increase water recycling thereby minimising demand from external water supply sources
- increase water supply security by diversifying the approved water supply sources to include surface water from the Lachlan River.

The MOD 4 application as originally lodged with the NSW Department of Planning and Environment on 4 May 2017, proposed the following changes:

- a relocation of the mine infrastructure area
- an increase in the Tailings Storage Facility footprint and height
- production of ammonium sulphate
- an increase in the annual production limit of sulphuric acid
- an increase in annual sulphur consumption

- an increase in limestone consumption
- allowing the possible backfilling of the eastern pit
- allowing access to site water sources for construction
- allowing extraction of surface water from the Lachlan River
- allowing provision of a water treatment plant
- minor changes to approved transport sources and transport methods.

Following further investigations into project optimisation, on 27 September 2017 Clean TeQ lodged a proposed variation of the MOD 4 application with the NSW Department of Planning and Environment. The proposed variation includes:

- no proposed relocation of the processing plant
- no backfilling of the eastern pit
- no increase in the annual production limit of nickel and cobalt
- no increase in the annual production limit of scandium oxide
- a minor reduction in the originally proposed limestone demand from 1,000,000tpa to 990,000tpa
- a reduction in the originally proposed ammonium sulphate production rate from 200,000tpa to 100,000tpa
- a change in layout of the proposed tailings storage facility
- the addition of a water pipeline realignment option in the vicinity of Fifield.

The MOD 4 application was made under the Part 3A transitional arrangements (Schedule 6A, clause 12) of the EP&A Act.

As at 30 April 2018, MOD 4 is with the Department of Planning & Environment for determination. Clean TeQ is preparing a response to a request by the NSW Department of Planning and Environment for more information regarding the MOD 4 application.

20.2.2.6 Modification 5

On 15 September 2017 Clean TeQ lodged a MOD 5 application (under the Part 3A transitional arrangements) with the NSW Department of Planning and Environment requesting approval to confine the timing and scope of the required Pre-Construction Hazard Studies (which under MOD 3 applied to further construction of any component of Clean TeQ Sunrise) to the Mine and Processing Facility and gas pipeline only (being the potentially hazardous project components).

The MOD 5 application was approved by the authorised delegate of the Minister for Planning on 20 December 2017.

20.2.2.7 Modification 6

On 22 December 2017 Clean TeQ lodged a MOD 6 application (under the Part 3A transitional arrangements) with the NSW Department of Planning and Environment for relocation of the proposed construction camp from the main project area to the off-Mining Lease Sunrise property (located 4 km south of the main project area), which will improve amenity for construction camp inhabitants and minimise operational constraints.

The MOD 6 application was approved by the authorised delegate of the Minister of Planning on 25 May 2018.

20.2.2.8 Future Modification Applications

The NSW Environmental Planning and Assessment Amendment Act 2017, which commenced on 1 March 2018, ended the Part 3A transitional arrangements that allowed the modification of the Project's 2001 Part 4 Development Consent under section 75W of the Environmental Planning and Assessment Act 1979. Any future modification applications will have to comply with the new legislative requirements under Part 4 of the Environmental Planning and Assessment Act 1979."

20.2.3 New South Wales Protection of the Environment Operations Act 1997

As at 30 April 2018 Clean TeQ is researching possible conditions for the required Environment Protection Licence under the NSW Protection of the Environment Operations Act 1997.

Clean TeQ expects to consult with the NSW Environment Protection Authority on suitable conditions of an Environment Protection Licence for the Project.

20.2.4 New South Wales Mining Act 1992

Clean TeQ's wholly owned subsidiary, Clean TeQ Sunrise, has obtained Mining Leases which finalised the Project's longstanding Mining Lease applications[30], by previous Project owners.

Mining Leases ML 1769 and ML 1770 were respectively granted to Clean TeQ Sunrise by the Minister for Resources on 15 February 2018 and 16 February 2018. The term of each Mining Lease is 21 years.

As a condition of each Mining Lease, an approved Mining Operations Plan is required prior to commencement of any significant surface disturbing activities, including mining operations, mining purposes and prospecting.

A change to the Development Consent (for example, approval of a future modification application) would typically require the replacement of an existing Mining Operations Plan with a new Mining Operations Plan[31].

Clean TeQ Sunrise Pty Ltd has purchased freehold land in and around the project areas in order to facilitate the grant of Mining Leases and future development of the Project. As at 30 June 2018 Clean TeQ Sunrise:

- owns all freehold land in the main project area (ML 1770)
- owns the freehold land to be developed as the rail siding
- owns the freehold land for the proposed off-Mining Lease Construction Camp (the MOD 6 application)
- is negotiating the purchase of freehold land for the Fifield bypass road

20.2.5 New South Wales Water Management Act 2000

Under the MOD 6 Development Consent (Schedule 3, clause 26), Clean TeQ must ensure that it has sufficient water for all stages of the Project and if necessary adjust the scale of development on site to match its available water supply.

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[30] These former Mining Lease Applications (MLAs) were: MLA 113 of 10 August 1998, MLA 132 of 20 September 1999, MLAs 139, 140 and 141 of 10 December 1999, and MLA 162 of 27 September 2000.

[31] Department of Planning and Environment - Division of Resources and Geoscience, ESG3 Mining Operations Plan (MOP) Guidelines, page 5.

The availability of a reliable water supply is critical for the success of Clean TeQ Sunrise. Clean TeQ has obtained water access licences and works approvals under the NSW Water Management Act 2000 as an essential step towards meeting the raw water demand of the Project.

In 2017 Clean TeQ engaged Golder Associates Pty Ltd to conduct a Water Management Assessment as part of the MOD 4 application process.

A number of groundwater monitoring sites have been established at the mine site and surrounds. Generally, groundwater levels are 30-60 m below ground level and follow the surface topography, being highest in the western area of the Project.

Two recent groundwater level measurements (in December 2016 and June 2017) have been recorded at the monitoring sites (including logger installation). The standing water level ranged between 210 m AHD to 280 m AHD.

Hydraulic testing (falling head) has been conducted and analysed on five of the existing groundwater monitoring locations (GAM 06, GAM 07, GAM 11, GAM 12 and GAM 15) at the mine site. This hydraulic testing indicates that hydraulic conductivities are very low and the groundwater is typically low yielding.

It is predicted that the processing facility (which will be the major user of raw water) will require 2,960 MLpa of raw water. Although this prediction is within the licensed limit of 3,154 MLpa under Clean TeQ’s existing WAL32068, Clean TeQ is conscious of the need to diversify water sources to reduce the Project’s dependency on groundwater and improve water supply security.

The MOD 4 application proposes to improve water supply security for Clean TeQ Sunrise by including (subject to obtaining the necessary water access licences and water entitlements) the extraction of surface water from the Lachlan River.

Clean TeQ will aim to minimise the raw water demand for Clean TeQ Sunrise by utilising recycled and treated process water and other water collected on site (e.g. internal run-off collection at the minesite including harvestable rights and mine dewatering and advance dewatering).

20.3 Environmental Management

Clean TeQ will implement effective measures to manage environmental risks and potential environmental impacts of the Project.

Sections 20.3.1 to 20.3.4 give an overview of the required environmental management measures.

20.3.1 Environmental Management Strategy

The MOD 6 Development Consent (Schedule 5, clause 1) requires Clean TeQ to prepare an Environmental Management Strategy for the Project in consultation with the relevant authorities and the Community Consultative Committee and to the satisfaction of the Secretary of the NSW Department of Planning and Environment.

The Environmental Management Strategy must (among other requirements) provide the strategic framework for environmental management of Clean TeQ Sunrise.

20.3.2 Environmental Management System

In order to provide an integrated and structured approach to environmental management including the management of environmental risks and potential environmental impacts, Clean TeQ is developing an Environmental Management System for the Project.

The Clean TeQ Sunrise Environmental Management System will outline the organisational structure, responsibilities, practices, processes and resources for implementing and maintaining environmental objectives of the Project.

20.3.3 Environmental Plans, Strategies and Studies

In addition to the Environmental Management Strategy, Clean TeQ must develop specific environmental plans, strategies and studies to meet the conditions of the required environmental approvals, licences and permits. Table 20-2 lists environmental plans, strategies and studies that will be required for Clean TeQ Sunrise as at 30 June 2018.

20.3.4 Environmental Monitoring Programs

Clean TeQ will ensure its environmental management of the Project includes monitoring programs under the conditions of the required environmental approvals, licences and permits.

The required environmental monitoring programs under the MOD 5 Development Consent are:

- noise monitoring program (Schedule 3, clause 9(d))
- blast monitoring program (Schedule 3, clause 16(c))
- air quality monitoring program (Schedule 3, clause 23(d))
- surface water monitoring program (Schedule 3, clause 30(b))
- groundwater monitoring program (Schedule 3, clause 30(c))
- revegetation strategy monitoring program (Schedule 3, clause 33(d))
- biodiversity monitoring program (Schedule 3, clause 35(d))
- road upgrade and maintenance strategy monitoring program (Schedule 3, clause 43(b))
- traffic monitoring program (Schedule 3, clauses 45(b) and 45(c))
- rehabilitation monitoring program (Schedule 3, clause 57(i)).

Clean TeQ must conduct an annual review of the environmental performance of the Project for the previous calendar year (Schedule 5, clause 5 of the MOD 6 Development Consent). The annual review must include (among other requirements) a comparison of monitoring program results against:

- relevant statutory requirements, limits or performance measures/criteria
- monitoring results of previous years
- relevant predictions in the Environmental Impact Statement (as defined in the Development Consent).

Clean TeQ must also commission and pay for the cost of a full independent environmental audit of the Project within one year of commencement of the development and every three years thereafter unless the Secretary of the NSW Department of Planning and Environment directs otherwise (Schedule 5, clause 10 of the MOD 6 Development Consent).

20.4 Potential environmental constraints

The key permitting constraint identified by SRK for the Sunrise 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.

20.5 Summary and conclusions

The primary environmental consent required for project implementation has been granted.

SRK has no reason to expect that the secondary approvals required for the project would be refused.

Table 20-1: Summary of Environmental Approvals, Licences and Permits That Are or May Be Required

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
Status: Granted				
1	Development Consent - DA 374-11-00 as modified by: - Modification 1 (MOD 1) - Modification 2 (MOD 2) - Modification 3 (MOD 3) - Modification 5 (MOD 5) - Modification 6 (MOD 6)	NSW Department of Planning and Environment	Development Consent was approved on 23 May 2001 and activated with partial construction of borefield in 2006. The borefield area was established assuming a duty/duty/standby arrangement. As at 30 April 2018, one bore in each of the eastern and western borefield has been developed and the western bore is operating (for local use under a water purchase agreement). MOD 1 application was approved on 15 October 2005. MOD 2 application was approved on 6 January 2006. MOD 3 application was approved on 12 May 2017. MOD 5 application was approved on 20 December 2017. MOD 6 application was approved on 25 May 2018	NSW Environmental Planning and Assessment Act 1979
2	Exploration Licences (EL): - EL 4573 - EL 8561	NSW Department of Industry - Resources and Energy	EL 4753 granted on 17 August 1993 (expires 16 August 2018) with option to renew for an additional three years. EL 8561 granted on 11 May 2017 (expires 11 May 2020).[32]	NSW Mining Act 1992 (section 22)
3	Mining Leases: ML 1769 ML 1770	NSW Department of Planning and Environment - Division of Resources and Geoscience	ML 1769 for the limestone quarry and limestone processing facility was granted on 15 February 2018 for a term of 21 years (expiring on 15 February 2039). ML 1770 for the mine and processing facility (the main project area) was	NSW Mining Act 1992: (section 63 – grant of mining lease) (section 98 – grant of consolidated mining lease)

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[32] Weblink - NSW Department of Planning and environment – Division of Resources and Geoscience weblink: www.commonground.nsw.gov.au.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			granted on 16 February 2018 for a term of 21 years (expiring on 16 February 2039).	
4	Approval that the Project is not a Controlled Action	Commonwealth of Australia Department of the Environment and Energy (DEE). Previously Commonwealth of Australia Department of the Environment from September 2013 to July 2016 (DoE).	Decision by letter of 6 February 2001 that proposed action set out in the Schedule is not a controlled action. On 10 August 2015 the Commonwealth Department of the Environment (DoE) was provided with a letter providing an outline of the MOD 3 application. DoE indicated on 19 August 2015 that it considered that the MOD 3 application would not change the potential impacts on matters of national environmental significance and therefore the modified Project does not need to be re-referred under the Environment Protection and Biodiversity Conservation Act 1999.[33]	Commonwealth of Australia Environment Protection and Biodiversity Conservation Act 1999 (section 75)
5	Water Access Licences (WALs) and works approvals for groundwater supplies: WAL32068 including works approval 70CA614098 WAL28681 including works approval 70CA610184	NSW Department of Planning and Infrastructure – Water (DPI-Water)	WAL32068 was issued by DPI-Water on 14 September 2012 and is a continuing tenure. WAL32608 includes nominated works approval 70CA614098 (issued by DPI-Water on 14 September 2012 with an expiry date of 12 March 2026) for six water bores located on the following lots (using the Upper Lachlan Alluvial Groundwater Source, Upper Lachlan Alluvial Zone 5 Management Zone) for up to 3,154MLpa: Lot 6, DP 598735 Lot 7300, DP 1150524 (x2) Lot 24, DP 752106 Lot 103, DP 752106 (x2)	NSW Water Management Act 2000 (section 56). Water Sharing Plan for the Lachlan Unregulated and Alluvial Sources 2012 (commenced on 14 September 2012) (WAL32068). Water Sharing Plan for the NSW Murray Darling Basin Fractured Rock Groundwater Sources 2011 (WAL28681). Bore licences under the NSW Water Act 1912 were replaced by a single Water Access Licence (see Schedule 10, clause 17 of the NSW Water Management Act 2000).[35]

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[33] Source: MOD 3 Environmental Assessment, pages 6 and 36.

[35] Weblink - http://www.legislation.nsw.gov.au/#/view/act/2000/92/sch10

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			WAL28681 was issued by DPI-Water on 16 January 2012 and is a continuing tenure. WAL28681 includes nominated works approval 70CA610184 (issued by DPI-Water on 16 January 2012 with an expiry date of 15 January 2025) using the Lachlan Fold Belt MDB Groundwater Source (Lachlan Fold Belt MDB (Other) Management Zone), for up to 243MLpa under the Water Sharing Plan for the NSW Murray Darling Basin Fractured Rock Groundwater Sources 2011.[34]	
6	Community Consultative Committee (CCC)	NSW Department of Planning and Environment (DPE)	The first CCC meeting was held on 14 February 2006. By letter of 26 April 2012 (ref 9038789), the New South Wales Department of Planning and Infrastructure accepted Ivanplats Syerston Pty Ltd's request of 8 March 2012 to have the CCC placed in abeyance whilst the project was on care and maintenance, however the NSW Department of Primary Industries requested the CCC be reconstituted in accordance with condition 10.1 of the approval at least three months prior to the commencement of any further construction works for the Project. The CCC has been reconstituted as requested by the DPE. By letter of 30 June 2017 the DPE advised it had appointed Lisa Andrews as independent chair of the CCC. The first meeting of the reconstituted CCC was held in Condobolin on 10 October 2017.	NSW Environmental Planning and Assessment Act 1979

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[34] Golder Associates report, Appendix D to MOD 4 Environmental Assessment, page 19.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
7	Aboriginal Heritage Impact Permit (AHIP)	NSW Office of Environment and Heritage (OEH)	AHIP number C0003049 for Clean TeQ Sunrise was issued by OEH to Clean TeQ Sunrise Pty Ltd on 10 October 2017.[36] The duration of the AHIP is ten years unless otherwise suspended or revoked in writing.	NSW National Parks and Wildlife Act 1974 (section 90)
8	Native Title determinations	Commonwealth of Australia National Native Title Tribunal (NNTT) ()	As at 30 April 2018 there are no registered Native Title claims over the various components of the Project.	Commonwealth of Australia Native Title Act 1993
9	Land ownership	NSW Department of Planning and Environment - Division of Resources and Geoscience	Clean TeQ's wholly owned subsidiary Clean TeQ Sunrise Pty Ltd is the registered owner of the following properties: Folio Identifier AC 8657-13 ('Syerston', comprising Lot 6 DP754021 and Lot 8 DP754021 located in the main project area). Lot 10 DP754021 ('Syerston', located in the main project area). Lot 9 DP754021 ('Kingsdale', located in the main project area). Lot 7 DP754021 ('Slapdown', located in the main project area). Lot 24 DP752089 ('Westella', located in the limestone quarry area). Lot 281 DP610057 ('Westella', located in the limestone quarry area). Lot 352 DP629402 ('Westella', located in the limestone quarry area). Lot 17 DP 752086 ('Sunrise', being the location of the proposed off-Mining Lease Construction Camp under the MOD 6 application). Lot 39 DP 752117 ('Glenrock') for the proposed railway siding.	NSW Mining Act 1992 (section 62(6))

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[36] Under the MOD 6 application, a new AHIP is required – see item 18 in this Table.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			Clean TeQ Sunrise's ownership of these properties means that section 62 of the NSW Mining Act 1992 (which sets out requirements for the grant of a mining lease over the surface of land on which a dwelling house, garden or a significant improvement is situated) did not apply to these properties.	
Status: Pending (In Progress)				
10	Modification of Development Consents: Modification 4 (MOD 4)	New South Wales Department of Planning and Environment (DPE)	MOD 4 application to realise additional project efficiencies was lodged with the DPE on 5 May 2017. Clean TeQ subsequently lodged a proposed variation of the MOD 4 application with the DPE on 27 September 2017. The Environmental Assessment required in support of MOD 4 application was lodged with DPE on 10 November 2017.	NSW Environmental Planning and Assessment Act 1979 (section 75W as continued by Schedule 6A, clause 12).[37] Environmental Planning and Assessment (Savings, Transitional and Other Provisions) Regulation 2017 (Schedule 4).[38]
11	Water Access Licence(s) and works approval(s) for surface water supplies	NSW Department of Primary Industries - Water	MOD 4 application has requested approval to access surface water from the Lachlan River to provide greater water security for the Project. If the MOD 4 application is granted, an additional Water Access Licence(s) will be applied for and water allocation(s) purchased.	NSW Water Management Act 2000 (section 60I)
12	Voluntary Planning Agreements (VPAs)	Lachlan Shire Council. Forbes Shire Council. Parkes Shire Council.	VPAs are in draft form (in Appendix 3 of the MOD 5 Development Consent). Contributions will be required at various stages of Clean TeQ Sunrise as detailed in each draft VPA. Under the MOD 3 Environmental Assessment (EA), Clean TeQ will	NSW Environmental Planning and Assessment Act 1979 (section 93F)

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[37] In this chapter, references to provisions of the NSW Environmental Planning and Assessment Act 1979 are references to those provisions as numbered prior to 1 March 2018.

[38] Formerly, clause 8J(8)(c) of the NSW Environmental Planning and Assessment Regulation 2000.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			contribute to the proposed road upgrades for the Project based on recommendations of GTA Consultants (in Appendix A to the MOD 3 EA) and consultation undertaken with the relevant Shire Councils[39], consistent with the terms of the draft VPAs which the relevant Councils have supported in principle. Clean TeQ will also contribute to maintenance of the roads identified in the MOD 3 EA consistent with the terms of the draft VPAs.	
13	Land purchases	NSW Department of Planning and Environment - Division of Resources and Geoscience	Clean TeQ Sunrise was negotiating an option to purchase agreement for Lot 11 DP 752089 and Lot 12 DP 752089 ('The Troffs') (located in the Westella limestone quarry area). Negotiations have been put on hold as there has been a strategic decision to purchase limestone from a local third-party source. Clean TeQ Sunrise is negotiating the purchase of other parcels of land for the Fifield Bypass Road, with exchange and settlement of agreements anticipated in 2018.	NSW Mining Act 1992 (section 62(6)) for Lot 11 DP 752089 and Lot 12 DP 752089
14	Landholder agreements: easement agreements for parts of natural gas pipeline not within road reserves compensation agreements for Crown land Lot 7303 DP1148889 and Lot 7302 DP1148734 use of Crown land Lot 28 DP 752111, Lot 29 DP 752077, and Lot 7300 DP 1150524 land access agreements for water pipeline.	Landowners and leaseholders. Lachlan Shire Council. Forbes Shire Council.	Easement agreements with the following parties for parts of the proposed natural gas pipeline not contained within road reserves expired on 14 February 2006 and will be renegotiated as required: (Peter and Terrie L'Estrange for Lot 4 DP 752087, Lot 5 DP 752087 and Lot 28 DP 752087 Timothy and Suzzane Jones for Lot 27 DP 752087 Lot 10 DP 752087).	NSW Crown Lands Act 1989 (section 34A) for Crown land. NSW Conveyancing Act 1919 (section 88B) for Council road reserves.

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[39] MOD 3 Environmental Assessment, page 26.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			Compensation agreements will be obtained for the land within Fifield State Forest (part of EL 4573) Lot 7301DP 1148734. Crown land Lot 7303 DP 1148889 (146ha) reserved for Future Public Requirements and Lot 7302 DP1148734 (101.9ha) reserved for the preservation of native flora and fauna will be impacted by an open cut pit and waste rock emplacement. Crown land Lot 7303 DP1148889 and Lot 7302 DP1148734 within the Project approval area must be subject to a compensation agreement to be agreed and executed prior to any mining activity taking place on this land.[40] In relation to Crown Land Lot 28 DP 75211, Lot 29 DP 752077, and Lot 7300 DP 1150524, as this land appears to be required for utility infrastructure, a licence under section 34A of the Crown Lands Act 1989 should be sufficient to authorise the use of the land. The proposed water pipeline route follows road reserves in the Lachlan and Forbes Shires. Clean TeQ will be required to enter into an agreement for the granting of an easement by each of Lachlan Shire Council and Forbes Shire Council, in order to obtain access to the relevant road reserve for construction and maintenance of the water pipeline.	
15	Aquifer Interference Approval (for mine dewatering)	NSW Department of Primary Industries – Water (DPI-Water)	On 4 August 2017 Clean TeQ completed the purchase of 243Ml of Lachlan Fold Belt MDB groundwater source entitlements as part of the water sharing	NSW Water Management Act 2000 (section 91(3)).

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[40] Department of Planning and Infrastructure Lands & Natural Resources submission to MOD 3 Environmental Assessment (Attachment A).

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
			plan for the NSW Murray-Darling basin fractured rock groundwater sources. The entitlement purchase will allow for the DPI-Water requirement in the Aquifer Interference Policy that Clean TeQ holds sufficient entitlement to account for any water take in the open cut pits due to groundwater inflows.	NSW Water Management (General) Regulation 2011 (clause 33). NSW Aquifer Interference Policy.[41]
16	Environmental plans, strategies and studies	NSW Department of Planning and Environment. (DPE). NSW Environment Protection Authority.	See Table 20-2 for a list of required environmental plans, strategies and studies.	NSW Environmental Planning and Assessment Act 1979. NSW Protection of the Environment Operations Act 1997. NSW Mining Act 1992. NSW Pipelines Act 1967. NSW Work Health and Safety (Mines and Petroleum Sites) Regulation 2014.[42]
Status: Outstanding (Not Commenced)				
17	Environment Protection Licence (EPL)	NSW Environment Protection Authority	An EPL is required because the Project will involve mining for minerals as a premise based scheduled activity if it will disturb more than 4ha of land by: clearing or excavating, or constructing dams, ponds, drains, roads, railways or conveyors, or storing or depositing overburden, ore or its products or tailings.[43]	NSW Protection of the Environment Operations Act 1997 (Chapter 3)
18	Aboriginal heritage impact permit (AHIP) to replace AHIP number C0003049 (required in relation to the MOD 6 application)	NSW Office of Environment and Heritage (OEH)	OEH has by letter dated 6 February 2018 to the NSW Department of Planning and Environment in relation to the MOD 6 application, stated that a new AHIP is required.	NSW National Parks and Wildlife Act 1974 (section 90)

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[41] Weblink - http://www.water.nsw.gov.au/__data/assets/pdf_file/0004/549175/nsw_aquifer_interference_policy.pdf.

[42] Refer to Chapter 16 of this Definitive Feasibility Study for more information on Work Health and Safety legislation.

[43] Schedule 1, clause 29 of NSW Protection of the Environment Operations Act 1997.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
19	Certificates of Competence for mine and quarry[44]	NSW Department of Planning and Environment (Division of Resources and Geoscience)	-	NSW Work Health and Safety Act 2011. NSW Work Health and Safety (Mines and Petroleum Sites) Act 2013. NSW Radiation Control Act 1990 (for class 7 radioactive materials). NSW Explosives Act 2003 (for class 1 explosives).
20	Licences for Storage and Use of Dangerous Goods (Hazardous Chemicals)[45]	SafeWork NSW	-	NSW Work Health and Safety Act 2011 (Schedule 11 hazardous chemicals)
21	Licences for Transport of Dangerous Goods (Hazardous Chemicals)	NSW Environment Protection Authority. SafeWork NSW.	-	NSW Dangerous Goods (Road and Rail Transport) Act 2008. Australian Dangerous Goods Code (current edition 7.5 which applies from 1 March 2018).
22	Licences for Environmentally Hazardous Chemicals	NSW Environment Protection Authority	-	NSW Environmentally Hazardous Chemicals Act 1985 (section 28)
23	Registration of dams	NSW Interim Dams Safety Advisory Committee, until commencement of section 6 of the Dams Safety Act 2015. Dams Safety NSW, after commencement of section 6 of the Dams Safety Act 2015.	-	NSW Dams Safety Act 2015 (Schedule 2, clause 5, (which commenced on date of assent of Dams Safety Act 2015 on 28 September 2015). NSW Dams Safety Act 1978 (to be repealed on the commencement of section 54 of Dams Safety Act 2015, which is not yet in force other than Schedule 2, clause 5 above).
24	Licence to construct and operate the Natural Gas Pipeline (Note: A licence for the Water Pipeline is not required, see section 5(1)(d) of the Pipelines Act 1967)	NSW Department of Planning and Environment (Division of Resources and Geoscience)	Schedule 2, clause 14 of MOD 5 Development Consent requires the design and construction of the natural gas pipeline to be in accordance with the relevant Australian Standards, in particular AS2885 Pipelines – Gas and Liquid Petroleum, or its latest version.	NSW Pipelines Act 1967 (section 14)

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[44] Refer to Chapter 16 of this definitive feasibility study for more information.

[45] Refer to Chapter 16 of this definitive feasibility study for more information.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
25	Permit for dredging work or reclamation work in a waterway	NSW Department of Primary Industries (Fisheries)	Permits may be required under the NSW Fisheries Management Act 1994 for any dredging or reclamation work in any waterway associated with construction of the gas or water pipelines or where activities associated with the pipelines will or are likely to damage marine vegetation.[46]	NSW Fisheries Management Act 1994 (section 201)
26	Approval to undertake works in, on or over a public road	Lachlan Shire Council. Forbes Shire Council. Parkes Shire Council. Roads and Maritime Services (RMS).	RMS requires a formal agreement in the form of a works authorisation deed between the developer and RMS for the developer to undertake "private financing and construction" of any works on Henry Parkes Way.[47] RMS has stated "prior to commencement of construction works, the proponent is to contact RMS's Traffic Operations Coordinator to determine if a Road Occupancy Licence (ROL) is required". In the event that an ROL is required, the proponent will obtain the ROL prior to works commencing within 3m of the travel lanes in Henry Parkes Way.[48]	NSW Roads Act 1993 (section 138)
27	Permits to transport oversize loads on public roads	National Heavy Vehicle Regulator (Oversize Overmass Vehicles)	-	NSW Road Transport Act 2013 (Chapter 6). NSW Heavy Vehicle National Law (commenced 10 February 2014).
28	Approval to alter a heritage place or demolish, move, alter or damage an archaeological relic	NSW Office of Environment and Heritage	-	NSW Heritage Act 1977 (section 139)
29	Licence to install and operate a sewerage system	Lachlan Shire Council	-	NSW Local Government Act 1993 (sections 68 and 68A)[49]

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[46] Ivanplats Syerston Pty Ltd 2005 Feasibility Study Update, Volume 2, Section 11, page 48.
[47] MOD 3 EA Responses to Submissions Table E1-1, RMS-2.
[48] MOD 3 EA Responses to Submissions Table E1-1, RMS-4.
[49] Weblink - http://www.lachlan.nsw.gov.au/f.ashx/Council/CouncilForms/1.16-Sewage-Management-Application-Form.pdf.

Item	Environmental Approval, Licence or Permit	Issuing Authority	Requirements and Comments	Relevant Legislation or Instrument
30	Occupancy permit for non-forest uses in State forests: Mine and Processing Facility limestone quarry (if applicable)	Forestry Corporation of NSW	-	NSW Forestry Act 2012 (section 60). NSW Mining Act 1992 (section 263).
31	Electricity tie line (alternative to natural gas pipeline)	Essential Energy	Clean TeQ has determined the proposed gas pipeline (as authorised under the MOD 3 Development Consent) will not be pursued in the Initial Production Phase (as defined in the MOD 3 environmental assessment, page 1). Instead, Clean TeQ intends to contract Essential Energy to construct a public infrastructure electricity supply tie line to the Mine and Processing Facility from the Trundle substation. If an electricity supply tie line is not ready for use by the time power is required, Clean TeQ intends to use diesel generators to meet short term on-site power requirements.	NSW Electricity Supply Act 1995 (section 45). NSW State Environmental Planning Policy (Infrastructure) 2007 (clause 41) made under section 37 of the NSW Environmental Planning and Assessment Act 1979.

Table 20-2: Required Environmental Plans, Strategies and Studies

Condition Reference	Required Plans, Strategies and Studies	Timing and Consultation Requirements
Development Consent MOD 6, Schedule 3, clauses 52 and 52A	Pre-Construction Hazard Studies consisting of: (a) fire safety study for the Mine and Processing Facility and gas pipeline (b) final hazard analysis for the Mine and Processing Facility and gas pipeline (c) construction safety study for the Mine and Processing Facility and gas pipeline (d) hazard and operability study for the Mine and Processing Facility and gas pipeline.	Pre-Construction Hazard Studies must be prepared and submitted for approval prior to commencement of construction of the Mine and Processing Facility and gas pipeline (except for preliminary works). The hazard and operability study for the Mine and Processing Facility and gas pipeline must be conducted by a suitably qualified and experienced team and chaired by a suitably qualified and independent person, whose appointments have been endorsed by the Secretary of the NSW Department of Planning and Environment. Construction of the Mine and Processing Facility and gas pipeline must not commence until approval of the Pre-Construction Hazard Studies has been given by the Secretary.
Development Consent MOD 6, Schedule 3, clause 53 and 53A	Pre-Commissioning Hazard Studies consisting of: transport of hazardous materials study for the development[50] Emergency Plan for the development Safety Management System for the development.	The Pre-Commissioning Hazard Studies must be prepared and submitted for approval prior to commissioning of the Mine and Processing Facility and gas pipeline. The emergency plan must be developed in consultation with the NSW State Emergency Services, Rural Fire Service and Fire & Rescue NSW. Commissioning of the Mine and Processing Facility and gas pipeline must not commence until approval of the Pre-Commissioning Hazard Studies has been given by the Secretary.
Development Consent MOD 6, Schedule 5, clause 1	Environmental Management Strategy	Prior to carrying out any development under this Consent after 6 May 2017 in consultation with the relevant authorities and the CCC and to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 35	Biodiversity Management Plan	Prior to carrying out any development under this Consent after 6 May 2017, in consultation with NSW Office of Environment and Heritage and to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 40	Heritage Management Plan	Prior to carrying out any development under this Consent after 6 May 2017, to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 30	Water Management Plan, which must include: water balance Surface Water Management Plan Groundwater Management Plan.	Prior to carrying out any development under this Consent after 6 May 2017, in consultation with Department of Primary Industries - Water and NSW Environment Protection Authority and to the satisfaction of the Secretary.

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[50] In this chapter, ‘the development’ as referred to in the Development Consent is the Project.

Condition Reference	Required Plans, Strategies and Studies	Timing and Consultation Requirements
Development Consent MOD 6, Schedule 3, clause 23	Air Quality Management Plan	On submission of an application for an Environment Protection Licence, unless otherwise agreed by the Secretary and the NSW Environment Protection Authority.
Development Consent MOD 6, Schedule 3, clause 9	Noise Management Plan	Prior to carrying out any development under this Consent after 6 May 2017 to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 33	Revegetation Strategy	Prior to carrying out any development under this Consent after 6 May 2017, in consultation with the NSW Office of Environment and Heritage to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 43 and Appendix 5	Road Upgrade and Maintenance Strategy	Prior to carrying out any development under this Consent after 6 May 2017, in consultation with NSW Road and Maritime Services and Council, and to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 45	Traffic Management Plan	Prior to carrying out any development under this Consent after 6 May 2017, in consultation with the relevant road authority and to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 57	Rehabilitation Management Plan	Prior to carrying out any development under this Consent after 6 May 2017, to the satisfaction of the Secretary.
Development Consent MOD 6, Schedule 3, clause 16	Blast Management Plan	Prior to carrying out any blasting at the limestone quarry, to the satisfaction of the Secretary.
Environment Protection Licence	Pollution Incident and Response Management Plan	To be determined.
Mining Leases ML 1769 and ML 1770, clause 3	Mining Operations Plan	Prior to the commencement of significant disturbance activities, including mining operations, mining purposes and prospecting.
Pipelines Regulation 2013, clause 12	Pipeline Management Plan (for the gas pipeline)	The date occurring six months after the commencement of operation of the gas pipeline.
Work Health and Safety (Mines and Petroleum Sites) Regulation 2014, clause 24[51]	Principal Hazard Management Plan	Before mining operations are carried out at the mine that may give rise to the hazard.
Work Health and Safety (Mines and Petroleum Sites) Regulation 2014, clauses 26 and 88[52]	Principal Control Plans	To be determined.

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[51] Some principal hazard management plans manage environmental risks; for example, air quality or dust or other airborne contaminants, and fire and explosion. Refer to Chapter 16 of the Clean TeQ Sunrise Definitive Feasibility Study Report for information on health and safety management.

[52] Some principal control plans manage environmental risks; for example, explosions and emergencies. Refer to Chapter 16 of the Clean TeQ Sunrise Definitive Feasibility Study Report for information on health and safety management.

21 Capital and Operating Costs

21.1 Capital Costs

Clean TeQ appointed SNC-Lavalin Pty Ltd (SNC-L) to execute a Definitive Feasibility Study (DFS) including an AACE Class 3 Capital Cost Estimate (capital cost), for the proposed Project.

As part of their scope, SNC-L has developed a capital cost estimate for the mine process, mine infrastructure and non-process infrastructure.

The estimate of capital costs comprises all expenditures required by Clean TeQ for the detailed design, construction and commissioning of the process plant and associated facilities.

The level of engineering is semi-detailed engineering development at 10% to 40% complete to produce an estimate with a nominal accuracy of -10% to +15%. The estimate was based on detailed equipment lists and extensive material take-offs commensurate with the maturity level of the project definition deliverables.

The total estimated capital cost for the processing facilities and associated infrastructure for Clean TeQ is AUD1,987,788,103 (USD1,490,841,077). Unless otherwise specified a AUD to USD exchange rate of 1:0.75 has been applied.

The project consists of the following facilities:

Processing Facilities

The processing plant is designed to treat 2.5 Mtpa of material into the PAL circuit and produce a combined metal equivalent output of up to 30 ktpa with no more than 25 ktpa of Nickel in NiSO4 and no more than 7 ktpa of Cobalt in CoSO4. The process plant includes:

- ore leach area
- refinery
- reagents
- services and infrastructure
- Stage 1 of the Tailings Storage Facility, together with associated water storage, evaporation ponds and storm water management.

Support Infrastructure

- including warehousing, workshops, site administration and office buildings, gatehouse and weighbridge, security and medical facilities.
- laboratory
- sewage water system
- power systems and electrical distribution
- bulk fuel storage
- access roads
- accommodation village
- wash down bays and explosive storage facilities are within the Mining Scope.

Off-Site Infrastructure

- bore field

- overland pipeline
- rail siding and associated facilities
- off-site roads
- electrical grid connections.

Mining

- The mining costs were priced by Clean TeQ and issued to SNC-L for input into the estimate.
- This scope includes mine related infrastructure to support the Mining Contractor. The cost of the mining fleet was included in the Mining Contractors unit operating rates.

A summary of capital costs is presented below in Table 21-1.

Table 21-1: 2.5 Mtpa capital cost estimate

Major Area	Description	Total Cost (AUD'000)	Total Cost (USD'000)
	Direct Costs		
1000	Site Development	15,871	11,903
2000	Mining	23,179	17,384
3000	Ore Leach	330,777	248,083
4000	Refinery	198,404	148,803
5000	Reagents	184,049	138,037
6000	Services and Infrastructure	303,068	227,301
	Sub-Total Direct Costs (TDC)	**1,055,350**	**791,513**
8000	Indirect Costs: Including field indirects, spares and first fills, commissioning & pre-commissioning, Alliance delivery	394,216	295,662
	Sub-Total Indirects	**394,216**	**295,662**
9000	Owner's Team (Including Owners Cost)	160,491	120,368
	Sub-Total Owner´s Costs	**160,491**	**120,368**
9000*	Estimate Contingency – P50	219,409	164,557
	Sub-Total Contingency & Risk	**219,409**	**164,557**
	Fee	158,318	118,738
	Sub-Total Fee	**158,318**	**118,738**
	TOTAL INSTALLED COSTS (TIC)	**1,987,788**	**1,490,841**

21.1.1 Exclusions

The following items were excluded from the capital cost estimate:

- escalation beyond estimate base date of Q2, 2018, through to the project completion
- foreign exchange rate variations beyond 1 July 2018
- GST or VAT on imported goods
- no application for the Australian Industry Participation (AIP) Scheme
- Australian Goods & Services Tax (GST) for materials / services provided from Australia
- import duties

- corporate taxes
- additional land acquisition costs
- no allowance for operating spares
- sunk costs: cost of this DFS and previous studies, purchase of autoclaves and agitators and the transportation of the autoclaves from Goro to Port Pirie
- no allowance for any variation to scope from that outlined in the MTOs issued by SNC-L engineering team
- no allowances for deferred capital costs
- no allowance for changes to industrial relation laws
- finance and interest charges for project duration
- costs of environmental testing
- any environmental requirement not identified in this estimate
- premium time or double shifts outside of the assumed working hours of 63.25 hours per week
- the weather conditions are not of extreme proportions that may disrupt the continuance of safe work. No allowance for abnormal weather conditions.
- the mining costs make no allowance for the provision of potable and general water, power, communications and sewage
- no allowance has been made for soil remediation for any in situ contaminants if required
- no allowance for rock excavation or piling
- no allowance for unidentified ground conditions
- schedule acceleration or delays such as those caused by information beyond the control of the contractors
- no allowance has been included for extended periods of industrial unrest
- no allowance has been made for delay costs associated with obtaining statutory approvals (e.g. building or development approval) post tensioning of concrete slabs
- landowner costs such as disturbance and reinstatement
- no allowances in base estimate or contingency for changes to the design resulting from the planned additional test work program
- no allowance for projects risks or schedule risks
- mining fleet at mine
- pre-stripping
- market forces related to the following:
- the effect of related concurrent projects on the availability of construction labour and materials
- the net effect of the world demand on prime commodities such as steel, copper and pipe
- the availability of suitable fabrication shops
- fuel price variation.

21.1.2 Currency

The estimate is expressed in Australian Dollars (AUD) and US Dollars (USD) unless noted otherwise, assuming a AUD1.00: USD0.75 exchange. Other exchange rates are shown in Table 21-2.

Table 21-2: Foreign Exchange Rates

Currency	Currency code	AUD 1: X
Australian Dollar	AUD	1.00
Canadian Dollar	CAD	1.00
Chinese Yuan Renminbi	CNY	5.21
Euros	EUR	0.67
Japanese Yen	JPY	86.21
South African Rand	ZAR	10.04
UK Pounds Sterling	GBP	0.59
US Dollar	USD	0.75
South Korean Won	KRW	877.19
Singapore Dollar	SGD	1.06

No allowance has been made for variation in the rate of exchange between the estimate base date and the placement of order.

SNC-L obtained budget pricing from potential bulk commodity suppliers and contractors, original engineering manufacturers (OEMs) and suppliers.

Table 21-3: Estimate Currency Basis

Currency Code and Description	% of Total
AUD – Australian Dollar	79%
EUR - Euro	2%
USA – US Dollar	18%
TOTAL	**100%**

21.1.3 Escalation

The base cost estimate is denominated in AUD, Q2, 2018.

No allowance has been made in the Capital Cost Estimate for escalation. Escalation is included within the owner’s reserve.

21.1.4 Working, deferred and sustaining capital

the Project capital costs are forecast to be incurred over a 24-month period from construction start in January 2019 to completion in December 2020 and includes total sustaining capital expenditure of USD233M beginning in year two of operations.

All sustaining capital costs are presented in real terms as at December 2018 and are based on a P50 estimate ±15%. Sustaining capital begins to be expended in January 2021 and continues each year until FY41 before closure costs are incurred in FY46/47. Most of the sustaining capital relates to construction of the Tailings and Storage Facility and for mine rehabilitation.

Closure costs are estimated at AUD125M with additional processing decommissioning costs of AUD8.4M in calendar year 2046.

21.1.5 Taxes and duties

Goods and Services Tax (GST) is not included in the estimate.

21.1.6 Estimate accuracy

The Accuracy of the capital estimate of -10% to +15% within the accepted range of a DFS.

21.2 Operating costs

All operating and maintenance costs are presented in real terms as at December 2018 and are based on a P50 estimate (±15%). All costs are presented in three forms:

- USD/lb of contained nickel sold
- USD/lb of contained nickel net of cobalt, scandium and ammonium sulphate by-product credits.

Operating costs are expressed as an average for mine operations between Year 2 and 25.

In addition to the engineering estimates, several allowances have been made to determine the operating cost for each of the operational components. These allowances include overhead and administration for owner's team costs, operating insurances, information technology and other operating items.

Operating costs have been compiled from a variety of benchmark sources including:

- first principle estimates
- suppliers' budget quotations and;
- consultants' data derived from similar external projects.

The breakdown of operating costs included the following categories:

Mining Costs

Mining costs are expressed in USD/lb contained nickel (real) and include:

- mining labour
- mobile crusher feed costs
- mining contractor costs
- grade control
- drill and blast

Processing Costs

Processing costs are expressed in USD/lb Nickel (real) and include:

- labour
- power
- reagents
- maintenance costs
- operating supplies

- utilities (power and water).

Haulage and Port

Haulage and port costs are expressed in USD/lb nickel (real) and include:

- haulage (mine to port)
- storage at port
- ship loading

General and Administration Costs

Other costs are expressed in USD/lb nickel (real) and include:

- general management
- health, safety and environment
- finance, human resources and logistics
- town administration
- insurances

Nickel and Cobalt Royalty

- Nickel and cobalt royalty costs are expressed in USD/lb nickel (real) and include State Government royalties for nickel sulphate and cobalt sulphate revenue.

Ivanhoe Royalty

- Ivanhoe royalty costs are expressed in USD/lb nickel (real) and include royalties paid to Ivanhoe Mines.

Table 21-4 shows the operating costs categorised into C1 and total cash costs FOB classifications

Table 21-4: Estimated Annual Average Operating Cost (Years 2-25)

Area	Description	Estimate AUDM	Estimate USD/lb Ni	Estimate[53] USD/lb Ni - Co Credits
Mining	Owner's team labour	4.73	0.09	
	Crusher feed	1.18	0.02	
	Mining contractor	53.83	0.99	
	Grade control and assay	1.54	0.03	
	Drill and blast	0.83	0.02	
Sub-total for Mining		62.11	1.14	
Process Plant	Owner's team labour	45.90	0.84	
	Sulphur	57.30	1.05	
	Limestone	9.71	0.18	
	Ammonia	11.91	0.22	
	Quicklime	5.69	0.10	

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[53] Excludes scandium and ammonium sulphate by-product credits

Area	Description	Estimate AUDM	Estimate USD/lb Ni	Estimate[53] USD/lb Ni - Co Credits
	Other reagents	18.61	0.34	
	Maintenance cons. & shutdown lab.	10.39	0.19	
	Operating supplies	7.58	0.14	
	Power and water	14.20	0.26	
Sub-total for Process Plant		181.27	3.33	
General and Administration	Owner’s team labour	5.86	0.11	
	Outbound logistics	3.85	0.07	
	Town administration	0.74	0.01	
	Insurance	1.00	0.02	
Sub-total for General and Administration		11.45	0.21	
Total		**254.83**	**4.68**	**(0.92)**

Table 21-5: Operating Cost Assumptions (Years 2-25)

Life of Mine Costs (Post Ramp-Up Years 2 to 25)	USD/lb Nickel (Real)		USD/lb Nickel (Real) Net of Cobalt Credits	
	C1	Total Cash Costs FOB	C1	Total Cash Costs FOB
Mining costs	1.14	1.14	1.14	1.14
Processing costs	3.33	3.33	3.33	3.33
General, administration and other site overheads	0.14	0.14	0.14	0.14
Haulage and port	0.07	0.07	0.07	0.07
By-product credits				
Cobalt credits	-	-	-5.60	-5.60
Scandium oxide credits	-	-	-0.37	-0.37
Ammonium sulphate credits	-	-	-0.18	-0.18
Cost Subtotal (C1)	4.68	4.68	-1.46	-1.46
Nickel and cobalt royalty	-	0.41	-	0.41
Ivanhoe royalty	-	0.34	-	0.34
FOB cash costs	-	5.43	-	-0.71

21.3 Key Qualifications

- The estimate base date is 31 May 2018.
- The estimate is reported in real Australian dollars.
- All costs incurred up to 31 Dec 2020 have been included in the capital cost estimate.
- Training and mobilisation of operating personnel, first fills and initial maintenance and operating spares are captured in the capital cost estimate.
- The scope of the operating cost commences from 01 Jan 2021.
- For this estimate, the project life is 25 years from the date of start-up.

- Labour costs assume 56 hours per week for continuous shift staff, 7-day coverage shift staff, and 38 hours per week for non-shift staff, provided on an annual basis.
- The long-term cobalt price assumption is USD30/lb.

21.3.1 Exclusions

- Depreciation
- Tax
- Import duty and sales tax
- Royalties and compensation for land use
- Regulatory and licensing costs
- Bank charges and interest
- Additions or deletions associated with further test work
- Owner’s corporate costs
- Escalation beyond the estimate base date
- Working capital movements

21.4 Quantity Derivation and Costs Basis

21.4.1 Mining

21.4.1.1 Owner’s Team Labour

The owner’s team labour for mining includes:

- mine management
- engineering personnel
- geology personnel
- survey personnel
- dispatch and pit personnel.

Cost Basis

All roles were allocated against standardised salaries aligned to industry benchmarks, which were taken from Mercer remuneration reporting.

This basis includes an addition loading factor including:

- superannuation guarantee contribution
- worker’s compensation
- payroll tax
- training allowance
- annual leave loading
- long service leave levy.

21.4.1.2 Crusher Feed

Crusher feed costs have been provided by IQE Pty Ltd. The crusher feed cost is provided on a Life of Mine basis and applied pro-rata annually.

21.4.1.3 Mining Contractor

Mining contractor costs have been provided by IQE Pty Ltd on an annual basis. The costs have been based off a wet hire arrangement where a contractor provides the required fleet on a fully maintained basis (inclusive of fuel costs), maintenance and servicing facilities, management, operators and maintenance personnel with Clean TeQ Sunrise providing the management and direction of the selective mining process. A summary of the Mining costs over the initial 25 year mine life is shown below:

Table 21-6: Cost Model Mining Cost Summary by Mining Activity Summary

Mining Activity Summary	Unit/Unit Rate	Cost
Clearing, grubbing, topsoil stockpiling	AUD	14,410,479
Grade control	AUD	35,749,859
Drilling and blasting	AUD	17,880,096
Load and haul	AUD	1,047,875,868
Ore re-handle and crusher feed	AUD	141,051,278
Establishment	AUD	2,643,000
Mobilise/demobilise	AUD	804,000
Haul road and civil construction	AUD	1,238,689
Contractor margin	AUD	151,398,392
Subtotal	**AUD**	**1,413,051,661**
Clearing, grubbing, topsoil stockpiling	AUD/t material moved	0.04
Grade control	AUD/t material moved	0.10
Drilling and blasting	AUD/t material moved	0.05
Load and haul	AUD/t material moved	2.91
Ore re-handle and crusher feed	AUD/t material moved	0.39
Establishment	AUD/t material moved	0.01
Mobilise/demobilise	AUD/t material moved	0.00
Haul road and civil construction	AUD/t material moved	0.00
Contractor margin	AUD/t material moved	0.42
Subtotal	**AUD/t material moved**	**3.92**
Total	**AUD/t ore feed**	**20.45**

21.4.1.4 Grade Control and Assay

Grade control and assay costs have been estimated by Clean TeQ based on internal historical data. The costs have been based off a wet hire arrangement where a contractor provides drilling services. Grade control samples will be prepared and assayed onsite.

21.4.2 Process Plant

21.4.2.1 Owner's Team Labour

The owner's team labour in the process plant, inclusive of operations and maintenance activities includes:

- production management
- utilities personnel

- ore leach personnel
- refinery personnel
- laboratory personnel
- technical and metallurgical personnel.

21.4.2.2 Sulphur

Quantity Derivation

Sulphur is used on site predominantly to make sulphuric acid in Area 5100. A further small amount is used directly as a chemical reductant in Area 3200.

Quantities

Sulphur quantities have been derived using the Mass and Energy Balance which itself has been derived based on information contained in the Process Design Criteria and using the process configuration given in the Process Flow Diagrams.

Cost Basis

Sulphur costs include separate commodity supply and transport and logistic costs.

The commodity cost basis for sulphur reflects the Fertecon Analyst Informa UK Ltd Agribusiness Intelligence reports, assuming an average of the 2016-2025 Vancouver free on board (FOB) price core benchmarks as reported between August 2017 and May 2018.

The logistics cost was modelled based on data provided by SASCAN. This includes:

- shipping Vancouver to Newcastle
- port costs and charges
- rail line haul
- container transport rail transfer
- rail siding to mine and processing facility transfer

21.4.2.3 Limestone

Quantity Derivation

Limestone is used on site for acid neutralisation in both Area 3400 and Area 3500.

Limestone quantities were derived using the Mass and Energy Balance which itself was derived based on information contained in the Process Design Criteria and using the process configuration given in the Process Flow Diagrams.

Limestone solids specification were based on a grade of 90% $CaCO_3$.

Cost Basis

Limestone costs were estimated based on supply from Ezy Lime Pty Ltd (Ezy Lime) including royalty, direct and freight costs based on a binding Memorandum of Understand between Ezy Lime and Clean TeQ Sunrise (as Clean TeQ Sunrise Pty Ltd).

Direct costs were inclusive of, but not limited to, all direct costs associated with extraction and truck loading of limestone including labour, blasting and equipment.

Freight costs were based on a freight distance of 76 km, allowing a net load of 75t per truck.

21.4.2.4 Ammonia

Quantity Derivation

Ammonia is used for pH control in Areas 4300, 4400 and 4500.

Ammonia quantities were derived using the Mass and Energy Balance which itself was derived based on information contained in the Process Design Criteria and using the process configuration given in the Process Flow Diagrams.

Cost Basis

The price of ammonia was derived from evaluated responses to Requests for Proposals conducted by Clean TeQ during 2017 – 2018.

21.4.2.5 Quicklime

Quantity Derivation

Quicklime is used on site for acid neutralisation in both Area 3600 and Area 4200.

Quicklime quantities were derived using the Mass and Energy Balance which itself was derived based on information contained in the Process Design Criteria and using the process configuration given in the Process Flow Diagrams.

Quicklime solids specification are based on a grade of 86% CaO.

Cost Basis

The price of quicklime was derived from evaluated responses to Requests for Proposals conducted by Clean TeQ during 2017 – 2018.

21.4.2.6 Other Reagents

Other reagents have been split into three groups. The first group comprises:

- formic acid
- hydrochloric acid
- SMBS
- soda ash
- caustic
- flocculants
- oxalic acid
- hydrogen peroxide.

The second group of other reagents comprises:

- ion exchange (IX) resins
- solvent extraction (SX) diluent
- SX extractants

The third group comprises:

- sulphuric acid plant catalyst

Quantity Derivation

The quantities of the first reagent group were derived using the Mass and Energy Balance which itself was derived based on information contained in the Process Design Criteria and using the process configuration given in the Process Flow Diagrams.

The quantities of the resin reagents are based on a benchmarked annual consumption as explained in Process Design Criteria reference T12 (Annual inventory consumption: 25% Year 1, 20% Year 2, 15% Year 3+). The working inventory of each reagent has been calculated based on the relevant equipment sizing and required working volume.

The quantities of SX organic reagents (diluent and extractants) were calculated based on the following loss assumptions:

- for Area 4300
    - an operating organic extractant concentration of 14% v/v;
    - 80 mL total organic consumed per m3 of raffinate to account for losses associated with solubility, entrainment and crud;
    - 5 mL/h of diluent evaporation per m2 of settler area; to account for evaporative losses (diluent only).
- for Area 4400
    - An operating organic extractant concentration of 23% v/v
    - 80 mL total organic consumed per m^3 of raffinate to account for losses associated with solubility, entrainment and crud
    - 5 mL/h of diluent evaporation per m^2 of settler area; to account for evaporative losses (diluent only)
    - Calculated extractant consumption on this basis equates to 2.6 kg of Cyanex 272 per tonne of produced Co. This compares well against a typical range of 3 – 5 kg/t quoted by the reagent supplier, considering the higher PLS Co grade for this project compared to most (and therefore lower unit consumption)
- for Area 4500
    - An operating organic extractant concentration of 30% v/v
    - 120 mL total organic consumed per m^3 of raffinate to account for losses associated with solubility, entrainment and crud
    - 5 mL/h of diluent evaporation per m^2 of settler area; to account for evaporative losses (diluent only)

The quantity of the third group, sulphuric acid plant catalyst, was based on the vendor's technical proposal to provide the basis for the quantity of each catalyst stage (1,2,3) and the average lifespan of each catalyst bed (6,6,10 years respectively). The annual consumption has been calculated by the volume of catalyst divided by the lifespan of the catalyst. The catalyst will be changed during acid plant shutdowns (2 x annual consumption every 24 months).

Cost Basis

The prices of the first group, all other reagents, were derived from indicative quotations from suppliers, and evaluated responses to Requests for Proposals conducted by Clean TeQ during 2017 – 2018, except for resins.

The prices of the second group, resin reagents, applied in the model were determined following an assessment of commercially available ion-exchange resins against the requirements of the DFS flowsheet, which included both metallurgical testwork and a Request for Proposal process.

The pricing for the third group, sulphuric acid plant catalyst, was determined by dividing the first fill quantity, provided in the vendor's proposal, by the volume and then multiplying by the volumetric annual consumption.

21.4.2.7 Maintenance Consumables

Maintenance consumables were estimated by Clean TeQ and reflect maintenance planning strategies, which allow for major, minor and unplanned maintenance of all plant and equipment. Maintenance consumables costs were estimated for:

- pipework
- wear components
- valves
- pumps
- motors
- electrical
- tanks
- vessels
- maintenance contract labour
- lubricants
- fuel
- consumables
- safety consumables
- vehicles
- tooling
- specialist training
- travel.
- Maintenance consumables costs are estimated against these categories on an annual basis.

21.4.2.8 Operating Supplies

Clean TeQ has estimated the annual allowances for operating expenses for:

- general building running costs
- hardware, software, and associated general operating costs
- operations staff transport (buses)
- AmSul (ammonium sulphate) facility lease costs
- office rental costs – Parkes.

Cost Basis

General building running costs were estimated at a fixed annual cost for:

- building cleaning
- laboratory consumables
- building maintenance

- building power.

Hardware, software, and associated general operating costs were estimated for information technology infrastructure and hosting, applications, hardware and licensing, sundries and other incidental costs, including but not limited to:

- internet and wide area network connectivity
- hosting charges
- consumable media
- operational infrastructure support
- application licenses
- application support team labour
- minor application maintenance
- sundries
- incidentals.

Operations staff transport was estimated to cover six buses running return trips, for each day of operation, running from Parkes to the operating site, at a fixed cost per return trip.

AmSul facility lease costs were estimated at a fixed cost per annum, assuming utilisation of a local facility provider.

Office rental costs in Parkes, NSW were estimated at a fixed cost per annum.

21.4.2.9 Power and Water

Power and water costs include:

- purchased power (electricity purchased from the NSW grid)
- water (sourced from both river and bore)

Water costs are estimated for raw water and include:

- external power cost at the bore field
- permitting costs

Quantity Derivation

The quantity of electricity for purchase was calculated using a multi-step approach. This calculation was performed for each operating cost period using the associated operating costs mass and energy balances.

Firstly, the total plant electrical load was estimated by using the nominal operating load as specified in the plant electrical load list. This load was corrected for several process areas that will not always be operating at their nominal load. These areas were:

- Sulphuric acid plant
- Nickel crystalliser
- Cobalt crystalliser
- AmSul crystalliser

Secondly, the process LP steam demand was estimated using a similar approach (variable demand for the above areas; fixed demand for the remainder of LP steam consumption).

Thirdly, the electricity that will be generated on site by the Steam Turbine Generator was calculated based on net steam production from the Sulphuric acid plant.

The quantity of electricity for purchase was calculated as the difference between the total plant electrical load and site generated electricity.

The raw water requirement for each operating cost period was taken directly from the associated mass and energy balance.

Cost Basis

The purchased power quantity and external power price was assumed to remain stable in real terms for the life of mine.

The permitting cost for raw water was applied per megalitre and was based on charges applied by the New South Wales State Government on aquifer share holdings.

21.4.3 General and Administration

21.4.3.1 Owner's Team Labour

The allowance for owner's team labour under 'general and administration' includes:

- general management
- health, safety and environment personnel
- finance personnel
- human resources personnel

21.4.3.2 Outbound Logistics

Outbound logistics to Newcastle port was estimated for:

- nickel sulphate
- cobalt sulphate.

Quantity Derivation

The quantity of hydrated nickel sulphate was estimated based on the projected production of nickel metal, in tonnes per annum on an annual basis, converted to hydrated sulphate form at a rate of 4.48t of nickel sulphate per tonne of nickel metal.

The quantity of hydrated cobalt sulphate was estimated based on the projected production cobalt metal, in tonnes on an annual basis, converted to hydrated sulphate form at a rate of 4.77t of cobalt sulphate per tonne of cobalt metal.

Cost Basis

The logistics cost was modelled based on data provided by SASCAN. This includes

- port costs and charges
- rail line haul
- road transfer.

21.4.3.3 Town Administration

Town administration costs were estimated based on in-principle agreed Voluntary Planning Agreements developed between Clean TeQ Sunrise (as Clean TeQ Sunrise Pty Ltd) and the Forbes, Lachlan and Parkes Shire Councils during 2018.

Voluntary Planning Agreements include:

- annual community enhancement contribution
- annual road maintenance contribution
- road maintenance costs were detailed with respect to specific activities estimated in-principle by each party in the localities.

21.4.3.4 Insurance

Indicative insurance costs were provided by Marsh Pty Ltd.

22 Economic Analysis

The economic analysis of the Project based on 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 Project is estimated to generate discounted cash flows totalling USD1,392M based on a discount rate of 8% (post-tax and real), assuming full project funding by shareholder equity. The post-tax real, ungeared internal rate of return (IRR) of the project is expected to be 19.1% with payback in 4.3 years from the start of operations.

This analysis has been based on a number of assumptions which include:

- Mining via contract mining operations of various pits to a ROM pad;
- maximum autoclave feed rate of 2.5 Mtpa. Average life of mine feed into the ore preparation plant will exceed the autoclave feed rate due to the beneficiated reject streams
- initial capital expenditure of USD1,491M (real)
- sustaining capital of USD233M (real) in total, assumed to start in FY22
- operating expenditure of USD4.68/lb contained nickel in $NiSO_4$ (post ramp-up on a real basis, excluding by-product credits)
- average weighted pricing received for:
- nickel in nickel sulphate of USD8.0/lb (real) free on board (FOB)[54], inclusive of a Nickel Sulphate premium of USD1.0/lb (real)
- cobalt in cobalt sulphate of USD30.0/lb (real) FOB
- scandium oxide of USD1,500/kg (real) FOB
- AUD:USD = 0.75 exchange rate based on the current market transactions.

In addition to determining the expected financial outcomes a series of sensitivities were performed for changes in:

- product volumes
- operating expenditure
- capital expenditure
- product prices
- construction delays

The sensitivity results showed that the project’s cashflow returns are robust and can provide a sustained future economic return.

22.1 Inputs and assumptions

The valuation assumes a 25-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.

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[54] Incoterms 2010.

Table 22-1 summarises the key inputs and assumptions:

Table 22-1: Sunrise valuation model key inputs

Parameter	Unit	Quantity
Autoclave Throughput	tpa	2,500,000
Average Strip Ratio		1.25
Autoclave Average Grade (Years 2 - 11)		
- Nickel	%	0.85
- Cobalt	%	0.20
Autoclave Average Grade (Years 2 - 25)		
- Nickel	%	0.75
- Cobalt	%	0.15
Average Recoveries (25-year)[55]		
Ni	%	92.6
Co	%	91.2
Life of Mine (initial)	years	25
Long Term Price Assumptions		
- London Metals Exchange nickel	USD/lb	7
- Nickel sulphate premium	USD/lb	1
- London Metals Exchange cobalt	USD/lb	30
Exchange Rate (AUD: USD) – Life of Mine	1 : n	0.75
Discount Rate	%	8
Tax Rate	%	30 initial, reducing to 25 (2027)
Royalties		
NSW Government (prior to allowable deductions)	%	4
Ivanhoe Mines (prior to allowable deduction of NSW Government royalty)	%	2.5
Depreciation		Straight line
Capital		
Total Development Capital Cost	USDM AUDM	1,491 1,988
Sustaining Capital	USDM AUDM	233 310

24.1.1 Basis of Estimates and Assumptions

Project financial forecasts are subject to foreign currency rate exposure as market products are priced in United States of America dollar currency (USD) whilst most costs are priced in Australian dollar currency (AUD). Forecast market prices for nickel sulphate, cobalt sulphate and scandium oxide have been obtained from various market studies and bank/broker consensus pricing forecasts.

The Project will require an initial capital expenditure of USD1,491M and total sustaining capital of USD233M beginning in year two of operations. Initial capital expenditure includes all civil works and infrastructure required to operate the mine.

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[55] High Pressure Acid Leach (HPAL) to final product

The operating cost is inclusive of all activities required to excavate and process ore and transport final product from the project site to port. This includes all maintenance activity for infrastructure and support facilities. It also includes all management costs for both site operations and corporate overheads.

Overhead costs include a team to manage/oversee the required contractors. The capital and operating costs for the processing plant assume that the plant is owned and operated by Clean TeQ.

Depreciation (non-cash) expenses for each aggregated capital item is calculated using prescribed accounting depreciation rates with a straight-line methodology. For calculation of corporate tax payments, a reducing-balance methodology has been applied for tax depreciation of assets when calculating taxable income. All capital items were assumed to have a 15-year operating life with any residual capital value existing at the end of this period written-off from a tax perspective by the company.

Taxes

A 30% corporate tax rate reducing to 25% over the duration of the project has been applied, in line with proposed Enterprise Tax Plan No. 2 2017.[56]Clean TeQ's prior year tax losses are accumulated and used to offset tax payments in future.

For financial evaluation and taxation purposes the Project is assumed to be an incorporated legal entity. GST has been excluded from all cost estimates.

24.1.2 Discount Rate Assumptions

A post-tax, real discount rate of 8% has been applied for evaluation purposes.

24.1.3 Pricing Assumptions

Investment evaluation for Project has assumed a set of weighted average prices for each product:

- nickel sulphate = USD8.0/lb (contained nickel), inclusive of a nickel sulphate premium of USD1.0/lb
- cobalt sulphate = USD30.0/lb (contained cobalt)
- scandium oxide = USD1,500/kg
- ammonium sulphate = USD90/kg FOB.

The total revenue for life of mine is USD14,066M (real).

Base case pricing forecasts assumed in the definitive feasibility study have been listed in Table 22-2.

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[56] Treasury Laws Amendment (Enterprise Tax Plan No. 2) Bill 2017 was introduced to the Australian House of Representatives on 11 May 2017. If passed, it will progressively reduce the lower corporate tax rate to 25% for all corporate entities by 2026-27.

Table 22-2: Pricing Assumptions

Product	Units (Real)	Financial Year								
		2017	2018	2019	2020	2021	2022	2023	2024	LT
Nickel Sulphate	USD/lb	8	8	8	8	8	8	8	8	8
Cobalt Sulphate	USD/lb	30	30	30	30	30	30	30	30	30
Scandium Oxide	USD/kg	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Ammonium Sulphate	USD/kg	90	90	90	90	90	90	90	90	90

24.1.4 Foreign Currency Analysis

Sunrise has exposure to USD through the market price and sale of nickel sulphate, cobalt sulphate and scandium oxide products and exposure to AUD denominated operating costs and approximately 18% of capital expenditure required.

24.1.5 Escalation and Inflation

Financial modelling was principally undertaken in real terms USD (December 2018) with key price and cost input assumptions assumed to remain constant in real terms.

24.1.6 Working Capital

Working capital is determined by the payment terms for debtors and creditors. For the study purposes it has been assumed that debtors will make payment on 15 days, which approximates the net contract terms for sales to proposed clients. Creditors will be on 30-day payment terms.

24.1.7 Royalties

Royalties included in cost assumptions include both a state government royalty and a royalty to Ivanhoe Mines.

The New South Wales State Government (State Government) royalty rate for nickel, cobalt and scandium is 4% ex-mine value (value less allowable deductions). This has been calculated by deducting processing and administration costs from revenue, as well as deducting capital allowances and other specific deductions. The royalty rate is then applied to the net revenue calculated.

Royalty to Ivanhoe Mines is also calculated with a 2.5% royalty rate applied to revenue net of the NSW State Government royalty.

22.1.1 Discounted cash flow analysis

Cashflow modelling of the Project, summarised in Table 22-3, demonstrates 100% equity NPV of USD1,392M (at a post-tax, real discount rate of 8%) with total earnings before interest, tax, depreciation and amortisation (EBITDA) cash flows over the 25-year project life of USD8,600M. NPV calculations exclude pre-FID expenditure totalling USD173M (definitive feasibility study and development capital expenditure prior to December 2018).

The cash flow model generated an internal rate of return (IRR) of 19.1% on a post-tax, real basis.

Table 22-3: Valuation Summary Pre-Tax Net Present Value

Key Metrics	Unit	Value
Post-tax NPV at 31 December 2018, 8% (real)	USDK	1,392
Post-tax IRR (real)	%	19.11
Simple payback (years, real CF) from 01 Jan 21	Years	4.33

Figure 22-1 presents the annual and cumulative post-tax ungeared free cash flow generated by the project. Total free cash flow (ungeared, post-tax) generated by the project is expected to be at USD5,126M based on definitive feasibility study assumptions.

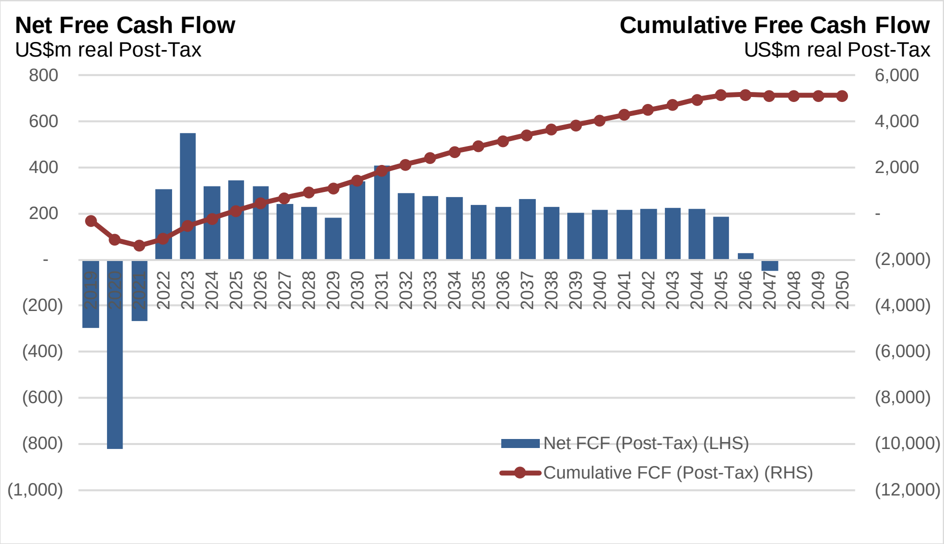


Figure 22-1: Net Free Cash Flow

Table 22-3 presents return on capital employed over the life of the Project. This metric has been calculated using the average opening and closing capital employed (total assets minus current liabilities) for each financial year. Return on capital employed has been defined as earnings before interest and tax (EBIT) divided by average capital employed in a period.

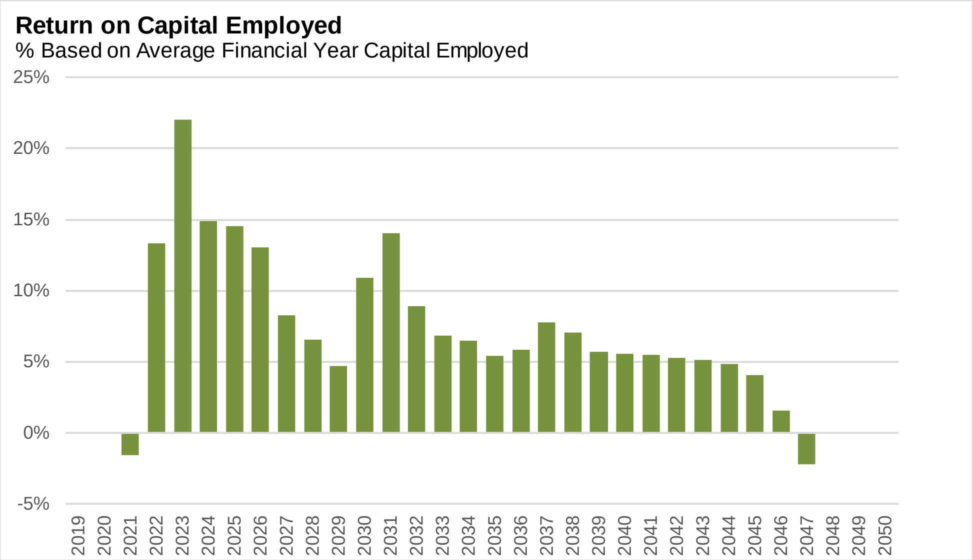


Figure 22-2: Return on Capital Employed

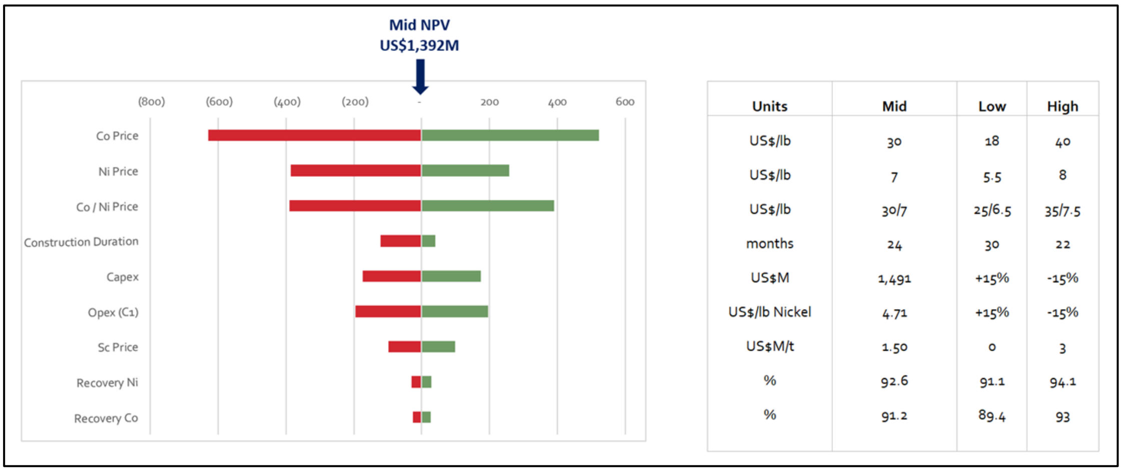
22.2 Sensitivity analysis

Multiple hypothetical scenarios have been considered in which selected critical assumptions are flexed within a given range. The sensitivity analyses presented in Table 22-4 to Table 22-11 and summarised in Figure 22-3 have been performed using the NPV result of USD1,392M (discounted at 8% post-tax, real) as the baseline comparison result.

Sensitivities were conducted on:

- product volumes
- operating costs
- capital expenditure
- product pricing
- impact of a delay in construction and subsequent operational readiness.

The sensitivity analysis calculated a positive NPV under all scenarios and provides an indication of a robust, economically viable project underpinned by strong cash flows.



Units	Mid	Low	High
US$/lb	30	18	40
US$/lb	7	5.5	8
US$/lb	30/7	25/6.5	35/7.5
months	24	30	22
US$M	1,491	+15%	-15%
US$/lb Nickel	4.71	+15%	-15%
US$M/t	1.50	0	3
%	92.6	91.1	94.1
%	91.2	89.4	93

Figure 22-3: Net Present Value Sensitivity Tornado Diagram[57]

Table 22-4: Nickel Recovery Sensitivity

Post Ramp-up Nickel Recovery	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
88.6%	1,827	1,302	906
89.6%	1,855	1,325	925
90.6%	1,883	1,347	944
91.6%	1,911	1,370	962
92.6%	1,938	1,392	981
93.6%	1,966	1,415	1,000
94.1%	1,980	1,426	1,009
94.6%	1,994	1,437	1,018

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[57] Nickel price shown excludes sulphate premium of US$1/lb.

Table 22-5: Cobalt Recovery Sensitivity

Post Ramp-up Cobalt Recovery	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
87.2%	1,853	1,322	922
88.2%	1,874	1,340	937
89.2%	1,896	1,357	952
90.2%	1,917	1,375	966
91.2%	1,938	1,392	981
92.2%	1,960	1,410	996
92.7%	1,970	1,419	1,003
93.2%	1,981	1,427	1,010

Table 22-6: Operating Expenditure Sensitivity

Operating Cost Sensitivity	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
-15%	2,111	1,532	1,097
-10%	2,054	1,486	1,058
-5%	1,996	1,439	1,020
0%	1,938	1,392	981
+5%	1,880	1,345	942
+10%	1,821	1,297	903
+15%	1,762	1,249	863

Table 22-7: Capital Expenditure Sensitivity

Capital Cost Sensitivity	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
-15%	2,109	1,560	1,147
-10%	2,052	1,504	1,092
-5%	1,995	1,448	1,036
0%	1,938	1,392	981
+5%	1,882	1,336	926
+10%	1,825	1,280	870
+15%	1,768	1,224	815

Table 22-8: Nickel Price Sensitivity

Nickel Price Sensitivity	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
-15%	1,555	1,082	725
-10%	1,681	1,184	809
-5%	1,810	1,288	895
0%	1,938	1,392	981
+5%	2,067	1,496	1,067
+10%	2,196	1,601	1,153
+15%	2,324	1,705	1,239

Table 22-9: Cobalt Price Sensitivity

Cobalt Price Sensitivity	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
-15%	1,652	1,157	784
-10%	1,745	1,233	848
-5%	1,842	1,313	915
0%	1,938	1,392	981
+5%	2,035	1,472	1,047
+10%	2,131	1,551	1,114
+15%	2,228	1,631	1,180

Table 22-10: Scandium Oxide Price Sensitivity

Scandium Oxide Price Sensitivity	Post-Tax Real Discount Rate and NPV Variance (USDM)		
	6%	8%	10%
-15%	1,920	1,378	969
-10%	1,926	1,382	973
-5%	1,932	1,387	977
0%	1,938	1,392	981
+5%	1,944	1,397	985
+10%	1,950	1,402	989
+15%	1,956	1,407	993

Table 22-11: Construction Delay Sensitivity

Construction Delay Sensitivity	Post-Tax Real Discount Rate and Variance (USDM)		
	6%	8%	10%
2 Quarters	1,826	1,270	853
1 Quarters	1,883	1,331	917
Current schedule	1,938	1,392	981

23 Adjacent Properties

Australian Mines Limited (AML) owns the Flemington project is reported (https://australianmines.com.au/flemington) … "to be a direction continuation … of the Sunrise project … with the two project separated only by a tenement boundary.

In May 2018, AML reported that:

"The Mineral Resource Estimate for the Flemington Cobalt-Scandium-Nickel Project is reported under JORC 2012 Guidelines and was reported by Australian Mines Limited on 31 October 2017. The Mineral Resource for Flemington, as announced on 31 October 2017 is, Measured 2.5 Mt @0.103% Co & 403 ppm Sc, Indicated 0.2 Mt @ 0.76% Co & 408 ppm Sc. There has been no material change or re-estimation of the Mineral Resource since this 31 October 2017 announcement by Australian Mines."

Source: https://australianmines.com.au/brochures/downloads/EGM_Presentation_-_May_2018_05_30.pdf

There are no resource projects/ properties immediately adjacent to the Project that are material to the findings of this Report. There are several large mining projects in the region as shown in Figure 23-1.

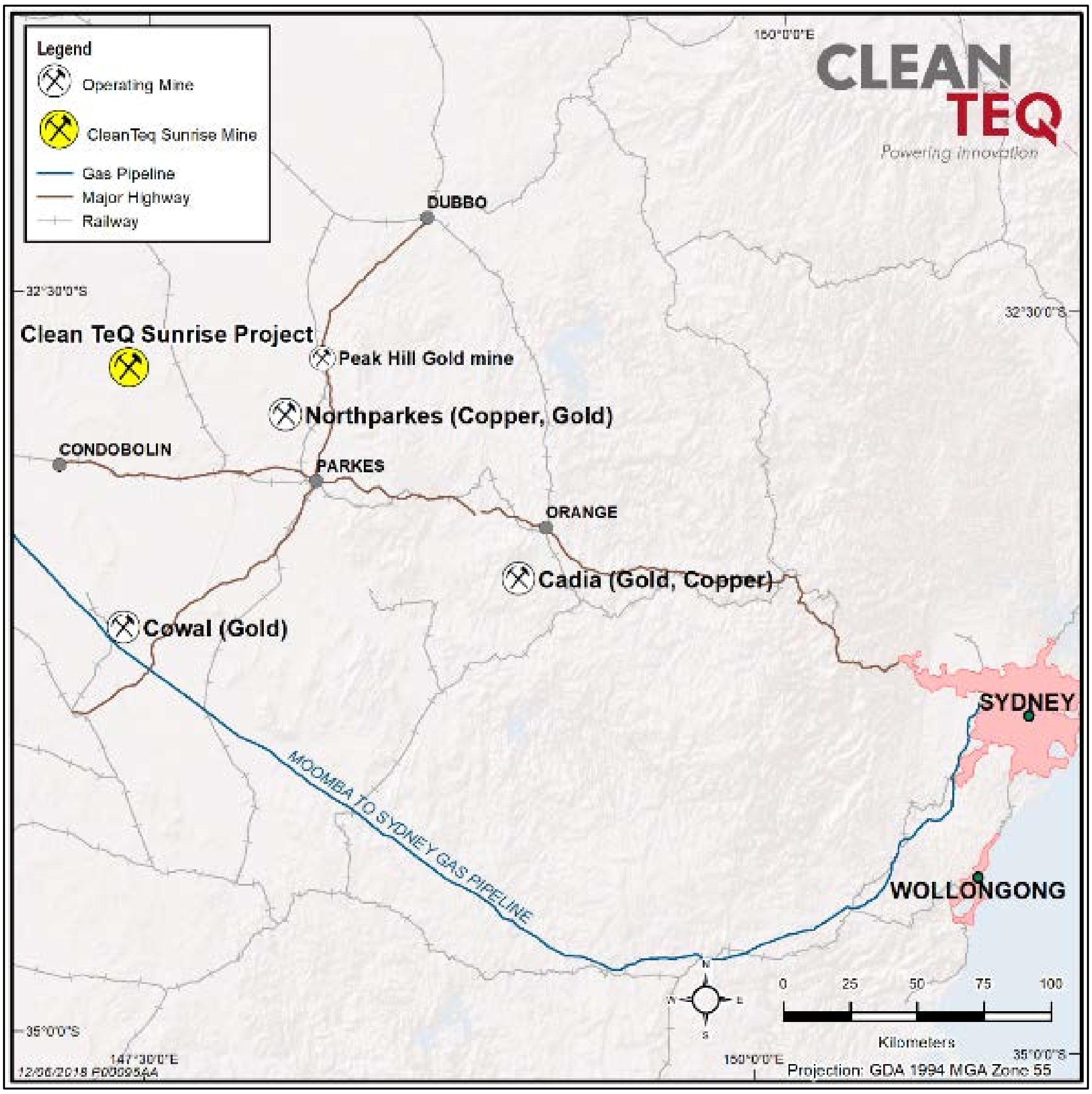


Figure 23-1: Other resource projects in the regional vicinity

24 Other Relevant Data and Information

This Report focuses on the extraction of nickel and cobalt and does not focus specifically on the extraction of Scandium.

Clean TeQ has capped production of Scandium at 10 tpa based on a price on USD 1,500/kg. This represents 3% of the total project revenue. Additional revenue from scandium is unclear and dependent on the establishment of a larger market and the impact on the commodity price.

The scandium is present in a 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 3 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 Sunrise typically followed scandium, as well as nickel and cobalt. All historical testwork confirms that scandium extraction using HPAL ranges from 80% - 90% and higher.

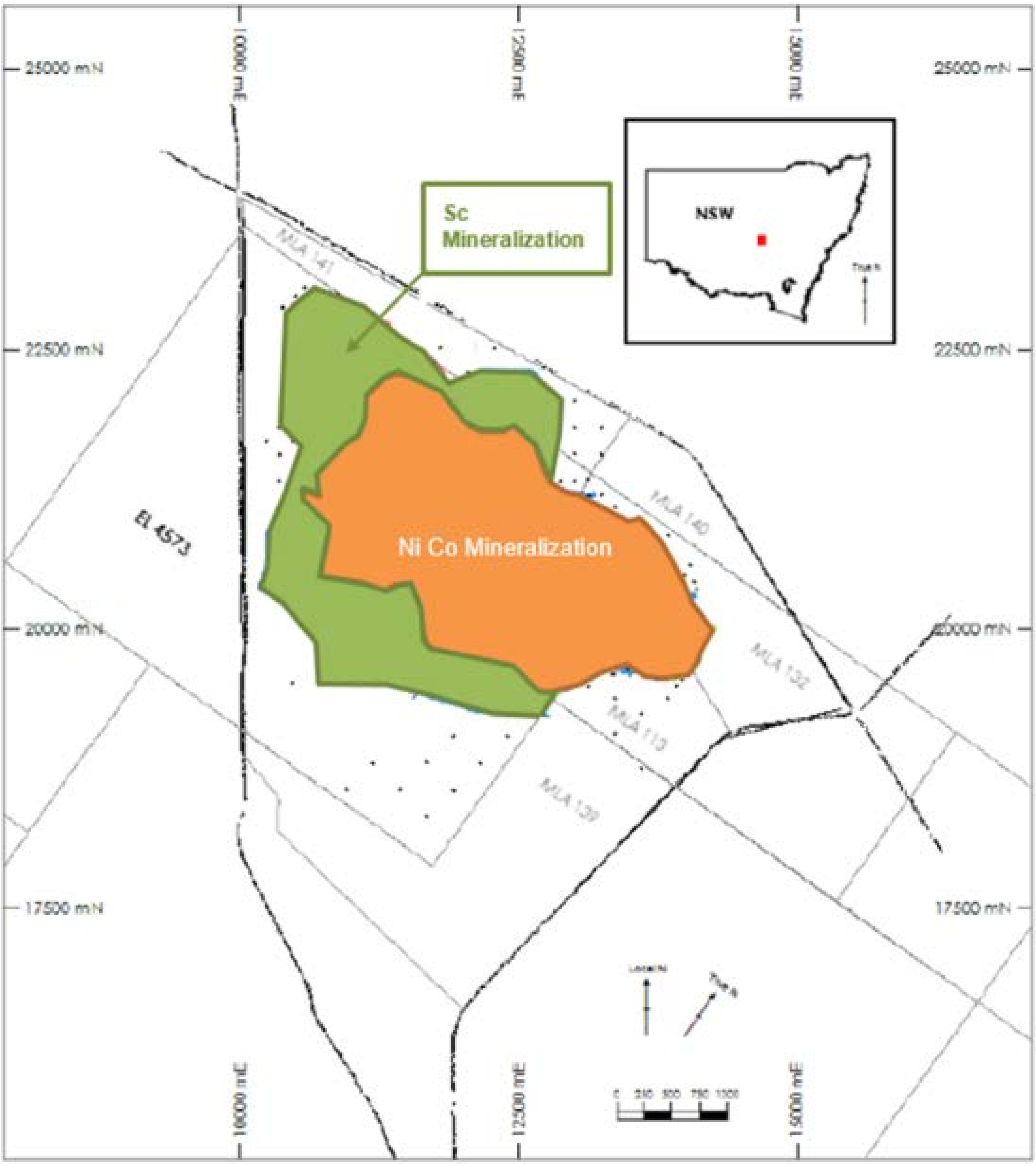


Figure 24-1: Scandium mineralisation location in relation to nickel cobalt mineralisation

25 Interpretation and Conclusions

Results of SRK's review of the 2018 Definitive Feasibility Study and supporting documentation, the QP's have concluded that the 2018 DFS summarises in this Technical Report contains adequate detail and information to support a Feasibility level analysis.

While no potential fatal flaws have been identified for the Project, the main areas of risk relate to project funding, execution systems, metallurgical assumptions and water availability.

The planned waste dump capacity of approximately 120 Mt is insufficient for the total waste production of 165 Mt. The 120 Mt capacity will be exceeded in year 15 of the project. Similarly, the planned medium to low grade ore stockpile capacity available is approximately 30 Mt, peaking at 93 Mt. The 30 Mt capacity will be exceeded in year 11 of the project.

Various options exist to increase the available capacity within ten years of project start-up and Clean TeQ is confident that one, or a combination, of the options will be successfully negotiated to solve this lack of capacity within the ten-year timeframe.

With regards to the metallurgical assumptions, having reviewed the 2018 DFS documentation and where appropriate other relevant information, and based on the information supplied, SRK concludes that:

- each of the individual flowsheet operations or steps selected in the overall process is technically and practically feasible
- the sequence of processing steps (the flowsheet) is technically and practically feasible
- the flowsheet selected is capable of producing the specified quality of products.
- the study has provided process and other engineering information such that all of the significant equipment selection and sizing can be undertaken

Clean TeQ have allowed for reduction in tonnes milled and metallurgical recoveries in the ramp-up period to mitigate the risk of achieving the predicted recoveries from the technical data. The assumptions were developed from data recovered when the various process steps were operating as well as could be expected at the pilot plant level. The metallurgical recoveries and plant throughput are presented in Table 25-1

Table 25-1: Process plant ramp-up Physicals

	Units	Yr 1	Yr 2	Yr 3 onwards
Nickel recovery	%	75.9	88.4	92.6
Cobalt Recovery	%	75.1	87.5	91.2
Mill Feed	tpmth	117	204	208

The metallurgical assumptions/ recommendations are discussed further in Section 26.

25.1 Risks

As part of the DFS, a risk assessment process used was consistent with ISO 31000:2009 Risk Management.

Attendees included:

- Clean TeQ personnel
- SNC-Lavalin Australia Pty Ltd personnel
- project execution personnel
- operations/maintenance personnel
- independent technical personnel
- mining consultant personnel.

The register contains, for each identified risk:

- risk assessment
- agreed risk treatment (mitigation) plan (where required)
- an individual assigned responsibility for ensuring the treatment plan is implemented
- specifies a due date for completion (where applicable).

Figure 25-1 and Figure 25-2 show both the risk profile at completion of the DFS, and the mitigated risk profile (i.e. anticipated profile at financial investment decision).

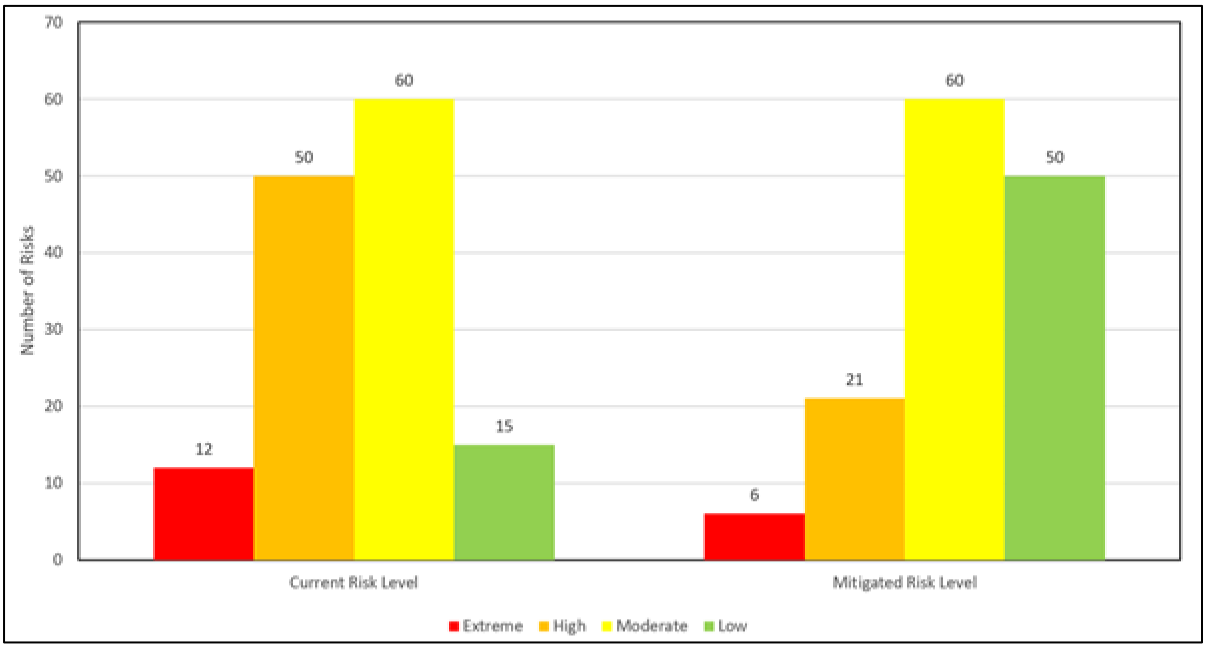
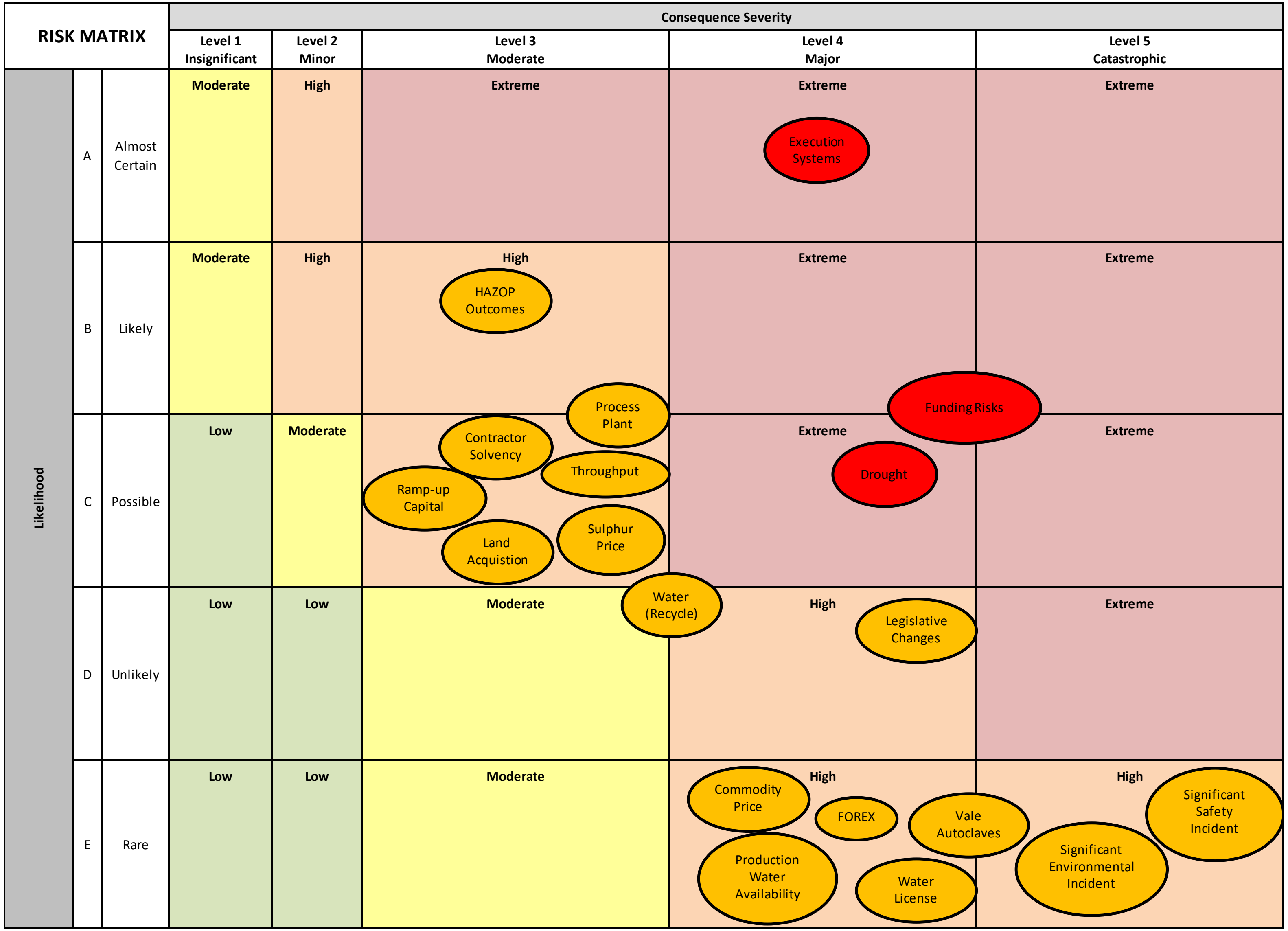


Figure 25-1: Risk reduction – from DFS to financial investment decision



RISK MATRIX			Consequence Severity				
			Level 1 Insignificant	Level 2 Minor	Level 3 Moderate	Level 4 Major	Level 5 Catastrophic
Likelihood	A	Almost Certain	Moderate	High	Extreme	Extreme Execution Systems	Extreme
	B	Likely	Moderate	High	High HAZOP Outcomes	Extreme Funding Risks	Extreme
	C	Possible	Low	Moderate	Process Plant Contractor Solvency Throughput Ramp-up Capital Sulphur Price Land Acquistion	Extreme Drought	Extreme
	D	Unlikely	Low	Low	Moderate	High Water (Recycle) Legislative Changes	Extreme
	E	Rare	Low	Low	Moderate	High Commodity Price FOREX Vale Autoclaves Production Water Availability Water License	High Significant Safety Incident Significant Environmental Incident

Figure 25-2: Clean TeQ Sunrise – Mitigated Risk Profile Bubble Diagram

25.2 Opportunities

A number of opportunities were identified in the DFS with the potential to improve the operability, performance and output of the processing plant, Table 25-2.

Table 25-2: Project risks and opportunities

Risk/ Opportunity	Summary
Technology	• The HPAL process is now in its fourth generation, with many of the learnings of previous operations incorporated in revised HPAL design work. • cRIP technology has successfully been implemented since the 1950s on over 30 full-scale operations to recover a range of metals. Clean TeQ has spent over 13 years developing and piloting the technology on laterite ores for optimal recovery of nickel, cobalt and scandium. • The resource and mining risk relating to the Project is assessed to be low because of the continuity of the deposit and the use of well-established resource estimation techniques. The reliability of nickel laterite resource and mining reserve estimates s improved with experience.
Political	• The primary Development Consent required for project implementation was granted in May 2017 and included permitting of water source and use (whether surface water or groundwater) to satisfy operational water requirements. • 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. • 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. • Mining Leases under NSW Mining Act 1992 for the main project area and the limestone quarry have been granted and are in good standing. • 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. • 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. • Land access/ acquisition agreements need to be finalised to cover parts of the Project area, limestone quarry, and rail siding. • Additionally, as a risk-mitigation measure, it would be desirable for the current water allocation to be increased, or additional Licenses purchased. • 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. Currently, cobalt supply is largely from Africa and nickel laterite developments tend to fall off rapidly in cobalt production capacity within the first five years of operation. To date, Clean TeQ Sunrise has secured offtake agreements for 20% of its Ni-Co production.

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. • 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: - recruiting and training the required numbers of suitably qualified and experienced management, supervision and operations personnel - establishing a suitable industrial relations structure for operations staff • The workforce establishment strategy emphasises recruitment of an experienced operations management team and to exploit knowledge and experience gained from currently operating lateritic nickel operations. Early operator and maintenance personnel training programs are to be adapted for local employees, and more experienced personnel from similar operations.

26 Conclusions & Recommendations

SRK's key recommendations relate to the metallurgical assumptions and ongoing testwork that will influence the project outcomes.

Autoclave feed rheology

Rheology testwork has indicated that the Sunrise ores exhibit variable rheology between 45%w/w and 48%w/w solids, with the yield stress often increasing with time (shear thickening).

It is recommended that further variability testing is undertaken to demonstrate that the currently proposed autoclave feed solids density of 45%, coupled with a coarser top size target of 500µm and elevated temperature (60 – 70C) will produce the desired rheology after going through the shear thinning process proposed in the plant design.

High pressure acid leach process

There is now a high degree of confidence in the robustness of the HPAL process to extract nickel and cobalt from lateritic ores, and extensive testing has been undertaken on the Sunrise ores. However, the current owners of the project have not demonstrated the high recoveries of nickel (95.7%) and cobalt (95.0%) that have been predicted at pilot scale, although former owner pilot testing and extensive batch testing by Clean TeQ has indicated these recoveries are possible.

It is recommended that the pilot plant operation commenced in July 2018 ensure that the nickel and cobalt recoveries in the process design criteria are confirmed at pilot scale.

Continuous Resin-in-pulp (cRIP) process

The process chemistry has been developed over many years now and has been proven in both uranium and gold operations. The testing undertaken by Clean TeQ has confirmed the resin chemistry for the current requirements and has not identified any issues that cannot be successfully managed. In addition, extensive pilot testing was undertaken with BHP to confirm the process suitability (both chemical and mechanical) for nickel laterite ores. Although further large-scale pilot testing is always desirable, this reviewer believes that the process is viable, and the process design criteria with regard to resin stability and replacement requirements are sufficiently conservative to cope with any unforeseen issues with regard to the mechanical and chemical stability of the resin.

One outstanding issue that remains is for the extent to which the cRIP process can recover nickel, cobalt and scandium that has been precipitated during the partial neutralisation process. A recovery factor of 20% for nickel and cobalt, and 40% for scandium, has been applied based on recent testwork that had issues with pH control in the cRIP circuit.

It is recommended that these percentage recoveries for all metals be confirmed for continuous processing using a resin pre-treated to achieve the correct pH balance in the cRIP circuit.

Downstream Processing

The extensive testwork undertaken by Clean TeQ over the last 18 months has mostly been enough to support the detailed design of the Sunrise Ni/Co Project. The design parameters are also supported by industry practice for nickel and cobalt solvent extraction operations which are well established.

The testwork adequately demonstrated that battery grade nickel and cobalt sulphates can be produced from the Sunrise ores, and the partial recovery of scandium, although only partially factored in to the process financial analysis, remains a project upside that may expand over time.

However, some of the cobalt purification processes (copper, zinc and manganese removal) were only tested on a relatively small scale in batch amounts.

It is recommended that further larger scale testing be undertaken on this process to better define the process design criteria and demonstrate their suitability for the Sunrise ore.

The scandium refining process has also only been demonstrated on a small-scale batch process. Further process development is required, as outlined in the Clean TeQ planned future testwork programs, to better define the flowsheet and process design criteria.

Product Quality

The testwork program has demonstrated that battery grade nickel and cobalt sulphates can be produced by the processes tested. However, a number of issues arose during the testing to indicate that unforeseen circumstances may arise during processing that will affect the product quality (contamination from a graphite coalescer and an undefined source of zinc contamination). It is understood the current plan is to sell off-specification material on the LME at discounted prices rather than re-treat the product. If this is possible, then it is a viable plan. If the off-specification material produced is more than expected, then the re-treatment option may need to be revisited.

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