Report for Altona Rare Earths Plc
Monte Muambe Competent Person's Report and Scoping Study
Project Number JB207282
April 2024

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1 EXECUTIVE SUMMARY

1.1 Background

This Competent Person's Report (CPR) and Scoping Study document has been prepared by Snowden Optiro, a business unit of Datamine Australia Pty Ltd (Snowden Optiro) for Altona Rare Earths Plc (Altona, the Company or the Client). This CPR has been prepared to provide compliant disclosure on all material mining assets and liabilities of Altona's Monte Muambe Rare Earth Element (REE) project (Monte Muambe or the Project), located in western Mozambique, in accordance with relevant Financial Conduct Authority (FCA) guidelines (FCA Technical Note 619.1 of May 2022, Appendix II), and the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 edition (JORC Code, 2012).

The Company is a UK London Stock Exchange (LSE) listed mining company focused on the supply of rare earth metal oxides for the catalyst, laser, glass, polishing and magnetic materials industries for the anticipated growth in the wind turbine and electric vehicle sectors.

The Project is an advanced exploration and pre-development project. Altona's interest in the Prospecting Licence (Licença de Prospecção e Pesquisa) LPP7573L (the Licence) is through a Farm Out Agreement dated 23 June 2021 between Ussokoti Investimentos, Altona, Monte Muambe Mining Lda (MMML) and its original shareholders. The Farm Out Agreement gives Altona the right to earn up to 70% of MMML in a phased manner, subject to the completion of certain conditions and milestones.

In this CPR and Scoping Study all references to dollars ($) refer to American United States dollars.

1.2 Project description, location and ownership

The Project is located in Niculunga Locality, Cambulassisse Administrative Post, Moatize District, Tete Province, Mozambique (Figure 1.1). Access from the provincial capital Tete is through the tarred road leading to the Malawi border (Zobue), through Moatize town, and to the village of Cateme. The distance from the Zambezi bridge in Tete to the Cateme turn-off is about 40 km. From the Cateme turn-off, an ungraded 43 km track leads from to the Monte Muambe camp through the villages of Mualadzi and Djendje. Figure 1.2 shows the location of Target 1 and Target 4 in relation to the Monte Muambe geology and structure and the proposed open pit outline (black).

Figure 1.1 Project location in Mozambique

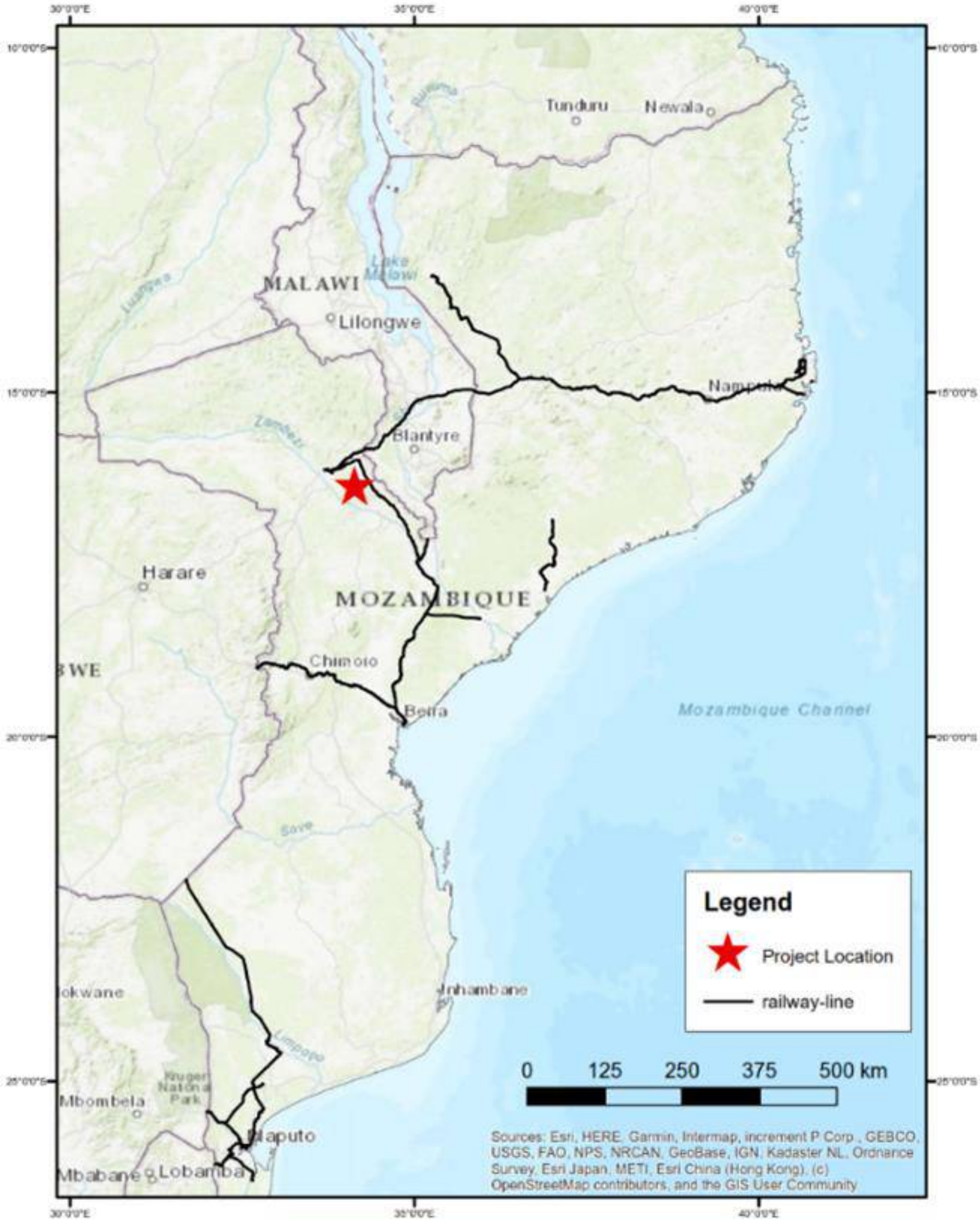
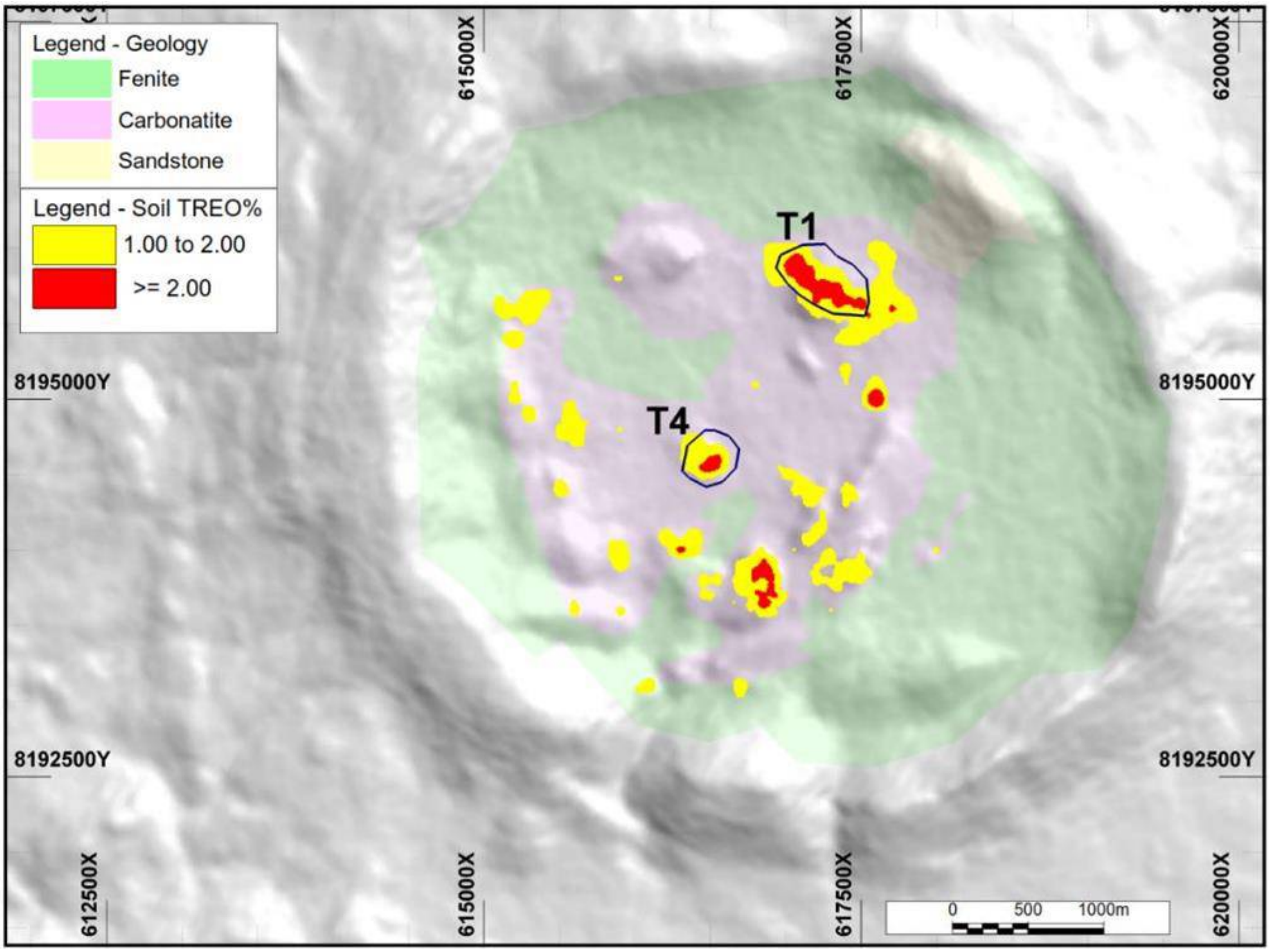


Figure 1.2 Location of Target 1 and Target 4 in relation to the Monte Muambe structure and geology



1.3 Geology and Mineral Resources

Monte Muambe is located in the central part of the Karoo Moatize-Minjova coal basin, which corresponds to the eastern part of the Zambezi Graben. The Monte Muambe carbonatite intrusion is hosted by Upper Karoo Sandstones of the Cádzi Formation. The age of the intrusion is presently unknown.

While the Monte Muambe structure resembles a ring-dyke, or a volcanic edifice, the outer ridge consists of sub-horizontal indurated Upper Karoo sandstones and is the product of differential erosion (Figure 1.2). The basin formed by the inner part of the structure consists chiefly of fenites, various types of carbonatites, breccias, as well as pyroclastics. The diameter of the carbonatite intrusion at surface level is about 3.3 km. Carbonatites tend to outcrop in the form of small hills rising above the floor of the basin. Fenites are often deeply weathered at near-surface levels and rarely outcrop, though float can be encountered on slopes.

Fenites form a circular zone lining the contact between carbonatites and host sandstones, but the detailed relationships between fenites and carbonatites are a lot more complex, involving faulting during and after the emplacement of the intrusion, as well as the incorporation of xenoliths of various size (centimetre to decimetre size). Drilling in various parts of the intrusion shows that fenite outcrops often cover carbonatites. This, as well as the presence of pyroclastics, suggests that the present erosion level may corresponds to the roof of the carbonatite intrusion, immediately under the base of the volcanic edifice.

The Mineral Resource outline is shown for Targets 1 and 4 (Figure 1.2). Mineral Resource Estimates for Targets 1 and 4 are provided in Table 1.1 and are reported in accordance with the JORC Code (2012).

Table 1.1 Monte Muambe Indicated and Inferred Mineral Resource September 2023 reported using a 1.5% TREO cut-off

Target	Classification	TREO Cut-off (%)	TONNES (Mt)	TREO%	CeO_2 ppm	Pr_6O_{11} ppm	Nd_2O_3 ppm	Tb_4O_7 ppm	Dy_2O_3 ppm	NdPr Oxide (ppm)	Contained TREO (t)
1	Indicated	1.5	8.0	2.38	11,400	910	2,250	15	80	3,160	191,000
	Inferred	1.5	0.8	2.28	10,900	861	2,140	15	78	3,000	18,000
	TOTAL	**1.5**	**8.8**	**2.38**	**11,400**	**905**	**2,240**	**15**	**80**	**3,150**	**209,000**
4	Indicated	1.5									
	Inferred	1.5	4.8	2.50	11,300	872	2,190	26	143	3,060	119,000
	TOTAL	**1.5**	**4.8**	**2.50**	**11,300**	**872**	**2,190**	**26**	**143**	**3,060**	**119,000**
OVERALL	Indicated	1.5	8.0	2.38	11,400	910	2,250	15	80	3,160	191,000
	Inferred	1.5	5.6	2.47	11,200	871	2,190	24	134	3,060	137,000
	TOTAL	**1.5**	**13.6**	**2.42**	**11,400**	**894**	**2,230**	**19**	**102**	**3,120**	**329,000**

Notes:

- *Million tonnes are rounded to one decimal place. Grades are rounded to two decimal places for % and whole numbers for ppm.*
- *The MRE has been reported in consideration of reasonable prospects for eventual economic extraction (RPEEE) using a pit shell based on a 1.5% Total Rare Earth Oxide (TREO) cut-off, revenue of 24.65 $/kg TREO in Mixed Rare Earth Carbonate (MREC) and average total recovery to MREC of 48%.*
- *Mineral Resources are reported as dry tonnes on an in-situ basis.*
- *Rare Earth Elements are inclusive of the TREO and not additional to it.*
- *"NdPr Oxide" is the sum of Nd_2O_3 and Pr_6O_{11}.*

1.4 Mining

The mining method is based on conventional open pit using truck and shovel, and drill and blast, coupled to a ROM stockpile. Although the rock is largely classified as weathered, ore and waste rock will require drilling and blasting.

Both ore and waste will be excavated in 5 m flitches following mark-out by grade control. Ore will be hauled to either the ROM pad and tipped onto a designated ore finger or a designated low-grade stockpile. All mine waste will be hauled directly from the pit and placed onto a designated location of the tailings storage facility (TSF) dam wall; there are no other external waste dumps.

The mining fleet will comprise 40 – 60 t capacity articulated dump trucks (such as a Caterpillar 745) loaded by a 90-t excavator (such as a Caterpillar 395). A 30-t front-end loader (Caterpillar 980M) capable of loading the 41-t dump trucks, will be used as back-up for the primary loading unit and to make up shortfalls in periods where additional material movement is required. Other ancillary support will be supplied by a Cat D9R dozer, Cat 14M grader, and Cat 745 watercart. Maintenance will be conducted on site. Contract-mining is selected as the operating strategy at the Project.

1.4.1 Pit optimisation

Optimization parameters used for the Scoping Study are summarised in Table 1.2 below. For Target 4 area the pit shell chosen was using the price increment of 102% which provided the highest undiscounted NPV as summarised in Figure 1.6. For Target 1 and 6, the chosen price increment was 98% which provided the highest undiscounted NPV (Figure 1.4).

The mine life is planned at 18 years. There is no pre-stripping period. Based on the selected pit shells a high-level pit design was produced.

Table 1.2 Parameters used in optimization

Item	Unit	Value
Total Rare Earth Oxide (TREO)	$/t	24,651
Royalty	%	3.0
Mining costs	$/t	3.28
Ore – free dig	$/t	4.26
Ore – drill and blast	$/t	2.51
Waste – free dig	$/t	3.53
Waste – drill and blast	$/t	3.28
Processing cost	$/t ore	25.00
Downstream processing cost	$/t TREO	66.00
Recovery from run of mine (ROM)	%	60
Recovery from refining	%	80
Throughput rate	Tonnes per year	750,000
Discount rate	%	10
Overall slope angle (OSA) T4	°	47
OSA T1 and 6	°	43

Figure 1.3 T4 optimisation result

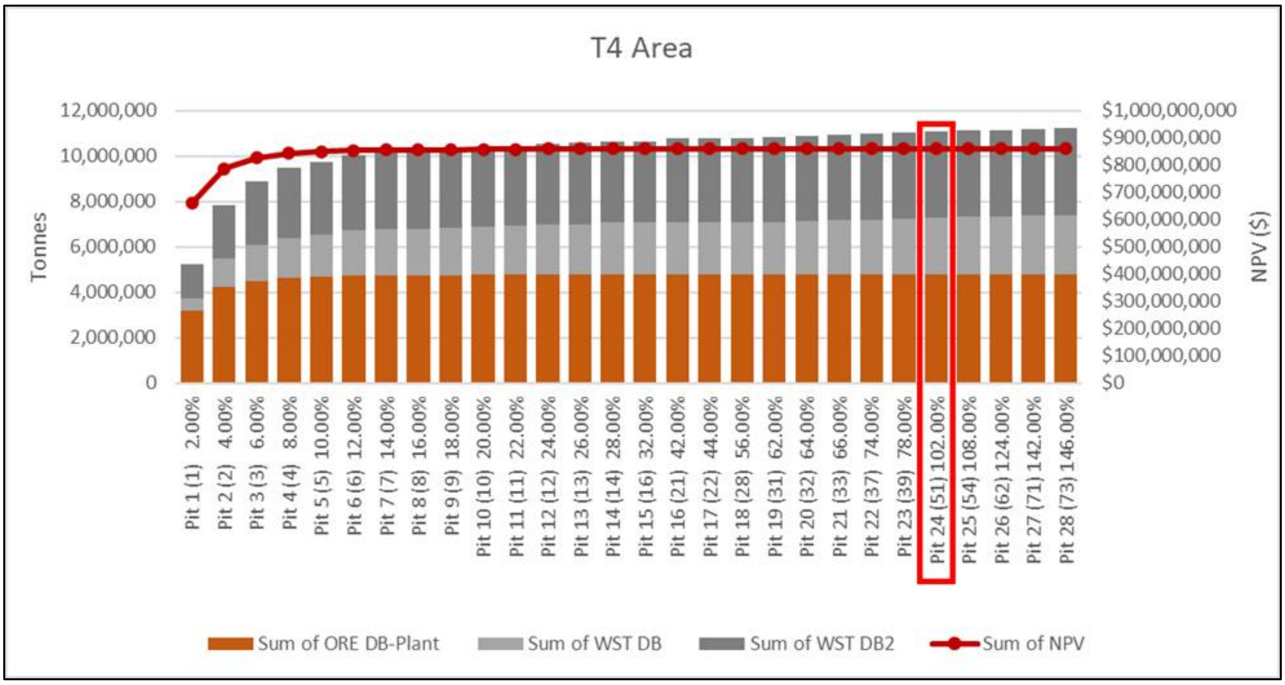
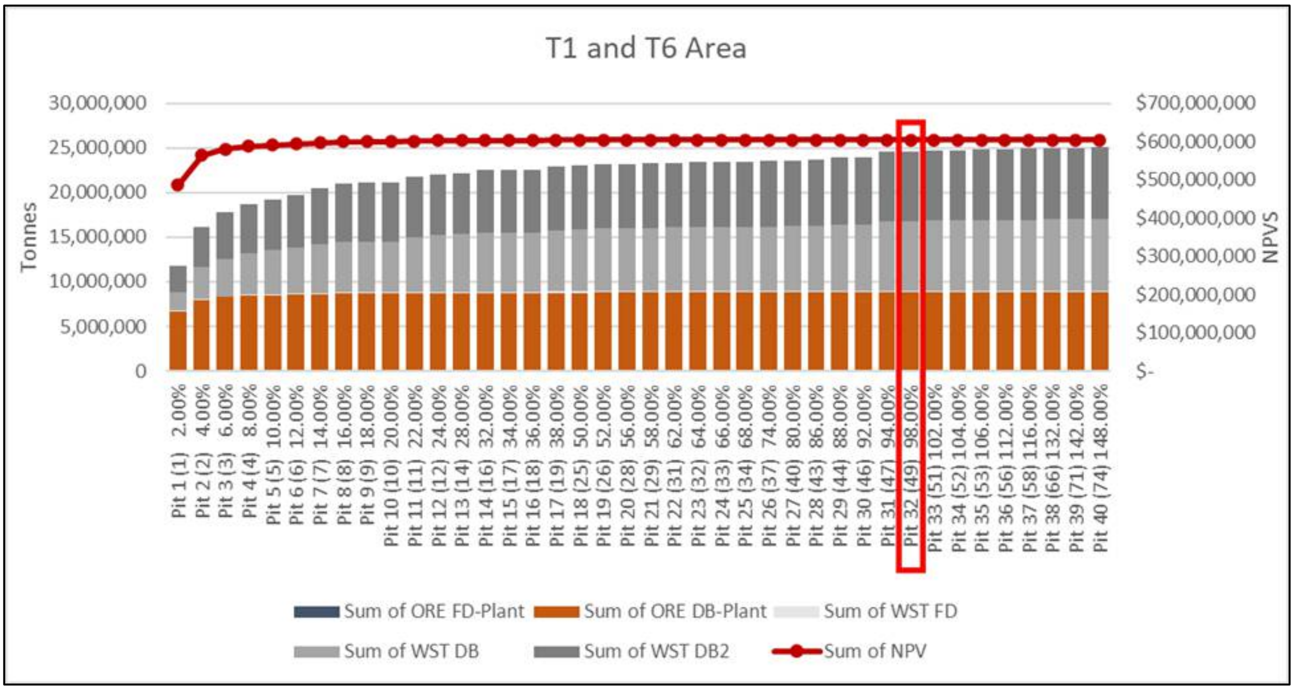


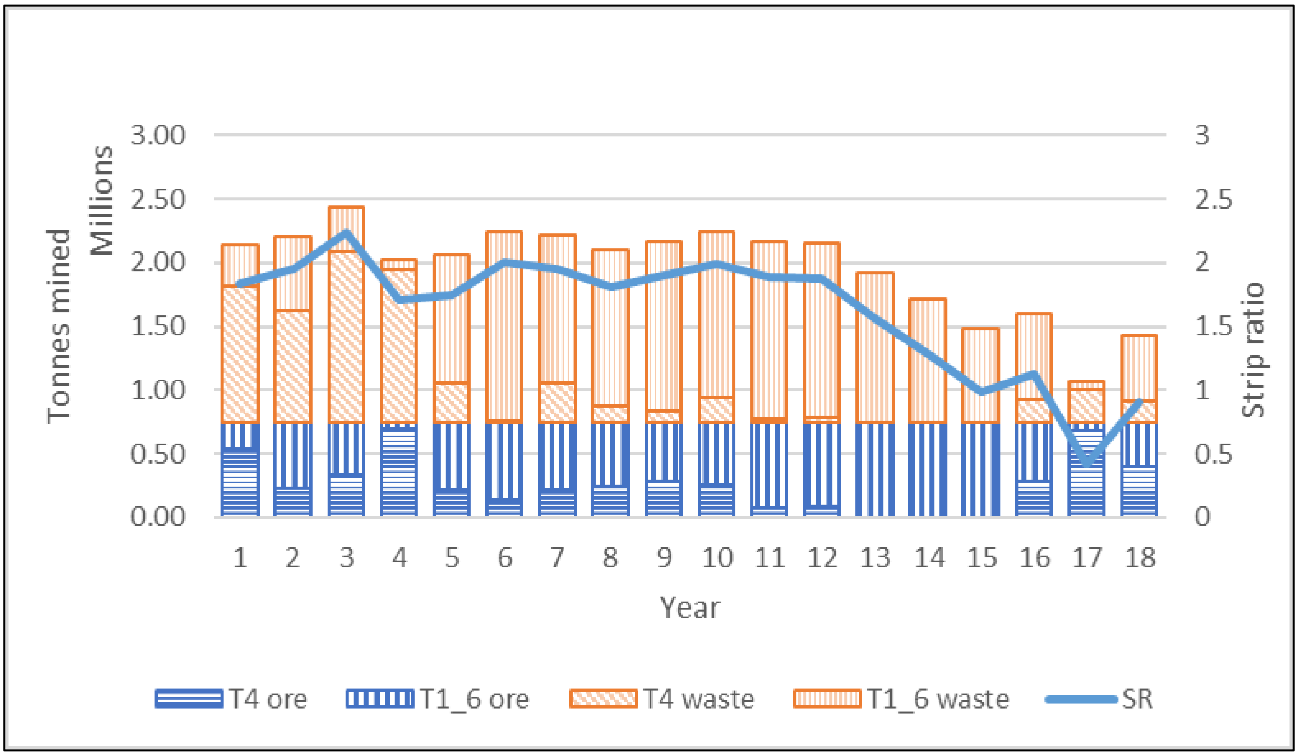
Figure 1.4 T1 and 6 optimisation result



1.4.2 Indicative schedule

An indicative life of mine (LOM) schedule was prepared for the mining of the two open pits as shown in Figure 1.5. Throughput rate is maintained at 750,000 t/a at a total mining rate of between 2.0 to 2.5 Mt/a. The average strip ratio is 1.67 (waste: ore). A pre-stripping period is not required but may be used to generate sufficient waste for the first TSF lift.

Figure 1.5 Monte Muambe annual LOM mining schedule



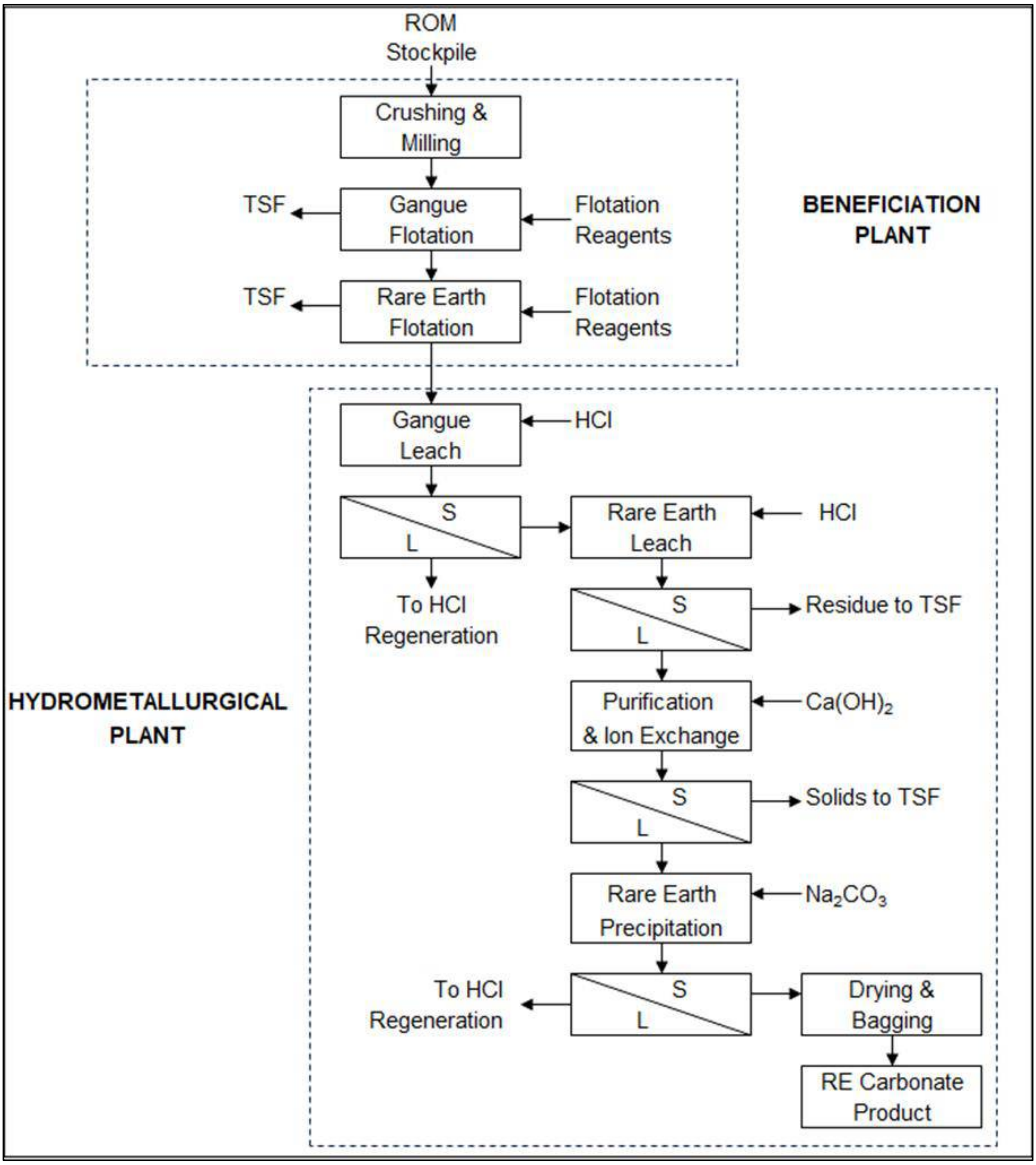
1.5 Metallurgy and processing

The proposed process flow sheet includes a beneficiation plant and a hydrometallurgical plant (Figure 1.6).

The beneficiation plant comprises of the comminution and flotation circuits. The purpose of the comminution circuit is to reduce the size of solid rock particles and thus increase the surface area of solids to enable the liberation of valuable materials that are locked within the gangue minerals. This is achieved by means of crushing and milling. Flotation is a method of separation, which uses the differing surface properties of the various minerals in the carbonatite. It involves the selective attachment of mineral particles to air bubbles generated in the flotation cell which float to the surface of the slurry and then flow over the lip of the cells into the launders. A two-stage selective flotation reagent regime is used; the first stage being a gangue flotation to selectively target the calcium bearing gangue minerals (calcite, fluorite and ankerite) and the second stage being a rare earth flotation targeting the host mineral bastnaesite.

The recovery process flow sheet comprises a two-stage selective hydrochloric acid leach process. The first stage being a calcite gangue leach in a weak (pH 4) acid solution and the second stage being a strong acid leach (20% HCl) at ~80°C. The hydrochloric acid is recycled via calcium sulphate precipitation with sulphuric acid which is produced on site via a commercial sulphur burner plant. The process flow sheet also includes purification and Mixed Rare Earth Carbonate (MREC final product) precipitation. This approach offers advantages, including a significant reduction in acid costs as well as a further concentration of the rare earths thus providing a reduction in downstream capital and operating costs.

Figure 1.6 Proposed process flow sheet



1.6 Project infrastructure

All required infrastructure, the accommodation camp, process buildings, stockpiles, water resources, and tailings storage facility (TSF) are located within the current mineral tenement boundaries. Site infrastructure is required both inside the crater and outside to service these facilities.

A conceptual site block plan (SBP) locates the main accommodation camp outside of the crater to minimize dust, noise and radiation exposure, whereas the process plant, mining contractors' workshops, TSF and associated infrastructure are deployed inside the crater, arranged to minimize the physical footprint and in close proximity to the two main pits.

The mine infrastructure and process areas are, as far as practical, consolidated to reduce materials handling distances, including that of run of mine (ROM) mineralisation. The SBP arrangement has considered the topography as well as accommodating future expansion of selected process units and exploitation of new pits.

Diesel generators will be used during construction and commissioning, and as backup for infrequent events of grid failure. It is assumed that the appointed bulk diesel supplier will install its own diesel storage tank(s) at site. Process, mine and office diesel storage tanks will be connected to the primary diesel storage tank(s).

The 18 MW electrical power maximum demand of the Monte Muambe site will be provided at 11 kV, 50 Hz by a hybrid power generation plant. The plant will comprise a diesel-powered electrical generator station and solar photovoltaic (PV) power station supported by a battery energy storage system. Cables will link the power station substation to the process plant substation.

Solar PV generation is expected to contribute approximately 25% to the overall generation. The solar PV power station will be contained in a separate area, 800 m upwind from the process plant.

Management consider that a bore well-field will be the most optimal solution for the Project’s water supply. The water demand for the Monte Muambe site will be supplied by on-ground overland pipelines from the bore field. Various sites for bore water have been identified and well tested. The bore field selected will provide water to the accommodation camp, process plant and camp. Dewatering from the open pits is also expected to increasingly contribute to the plant’s water supply during the life of the mine. A water demand forecast will be designed as part of the PFS.

Wastewater and overall water management will be achieved by suitably planned drainage channels and site layout. Waste will be collected in a landfill.

The Project is primarily accessed from the northern side. A tarred, single carriageway (N7) extends from Tete, passed Moatize coal mine in a north easterly direction, a total distance of 70 km. There is a right turn onto a tarred single carriageway extending in a southeasterly direction for 10 km to Cateme. From Cateme, a 35 km gravel road is used to access the Project site; this road passes through the villages of Mwaladzi and Dezemge (Figure 1.7).

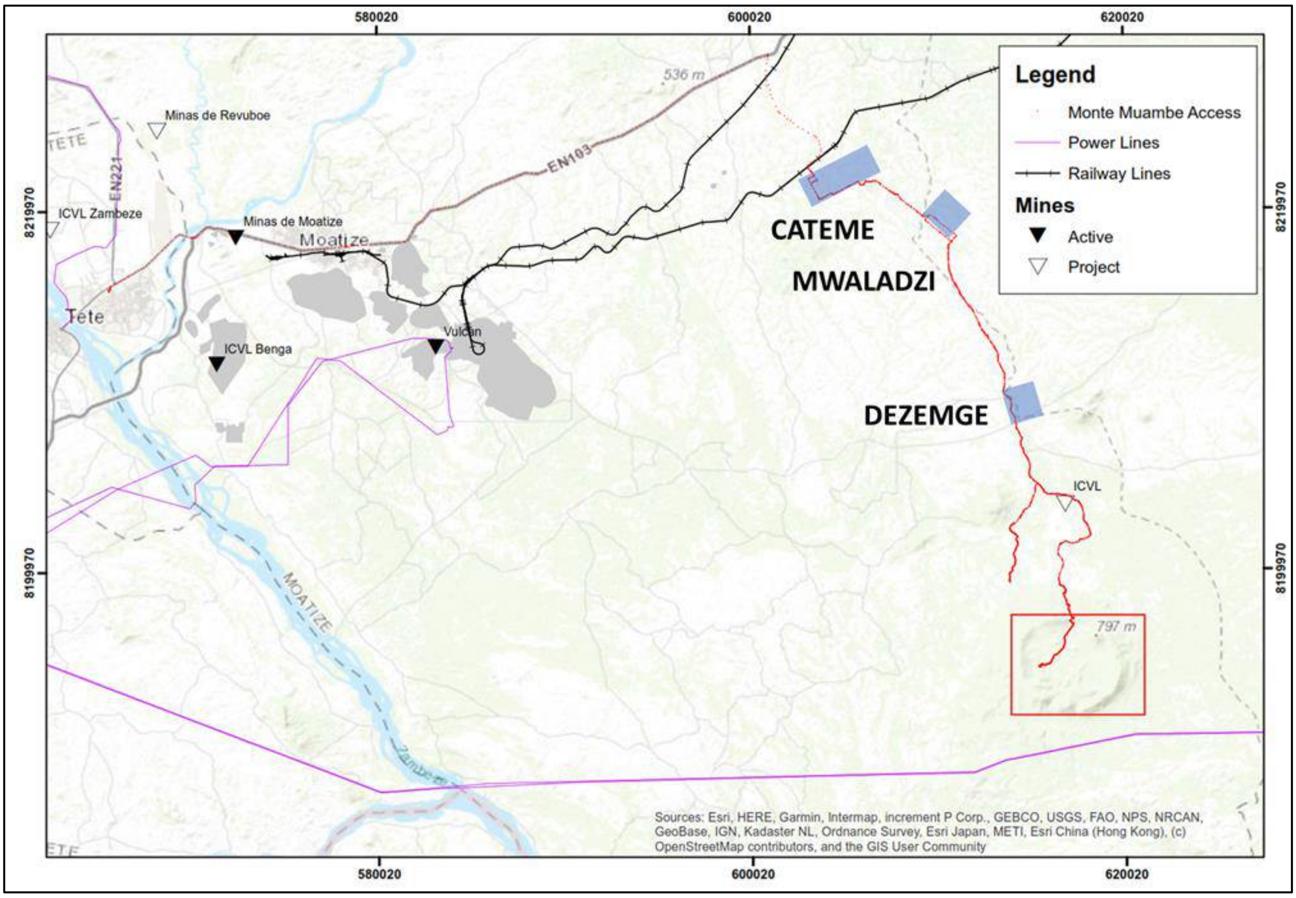
The road from Dezemge to the Project site will require upgrading, including by-passes around villages. The road climbing from the foot of the mountain to the existing camp and into the basin will need to be redesigned to ensure a maximum slope of 10 percent. Inside the basin, where the planned mining infrastructure and plant facilities will be located, the topography is gentle. Existing dirt tracks will require widening and upgrading using locally sourced road metal.

The Tete International Airport (Chingozi) or TET, is 110 km by road to the Project, along the N7. Beira is the closest port to the Project site, approximately 730 km by road. A detailed cost-benefit of the various logistical options both for inbound and outbound freight cargo will need to be done as part of the PFS.

An integrated information system will be provided by the Company, including the latest operating systems enabling effective telephonic and digital communications.

For product transport, it is proposed that stockpiled MREC will be placed in 1 t polypropylene, double-lined woven bulk bags at Project site, and then placed on pallets or loaded directly into containers. Containers will be trucked to Beira port and warehoused, prior to shipment. These transport arrangements are expected to result in approximately 745 truck journeys per annum (equivalent to 62 trips per month) of bagged concentrate product to Beira. The containerised bags will be offloaded at Beira and then re-containerised at Beira or report straight to ocean going vessels.

Figure 1.7 Location of the Project, licence LPP7573L in Tete Province, Mozambique



Source: Altona, 2023

The approximate infrastructure size and costs for the Project have been estimated. Primary infrastructure costs include:

- Power ($7.5 million)
- Access road ($7.0 million)
- Accommodation ($4.0 million)
- Sewage treatment ($2.0 million)
- Raw water dam ($2.0 million)
- Wellfield ($2.0 million)
- Stormwater ($1.0 million)
- Water treatment ($1.0 million)
- Other surface infrastructure (including gatehouse, changehouse, laundry, clinic, canteen, office buildings), of $2.8 million.

The approximate footprint of the ten primary surface infrastructure/ buildings is 10,915 m^2. As the Project advances, greater accuracy and footprint size will be estimated.

Design details will be required as the project advances to PFS stage; this will include as a priority:

- Power demand
- Water demand
- Detailed plans for site location
- Detailed access road plans.

1.7 Tailings and waste

1.7.1 Tailings storage facility

All process plant waste products will likely be disposed of onto a single fully contained tailings storage facility. Pre-stripping over the mining area will provide the initial waste rock required for the containment embankment walls. As more waste is stripped over the mining areas, these waste rock embankments will be raised always above the tailings level to provide solid rock embankment walls. The intention is to, where possible, use the existing topography and outcropping areas to buttress the final TSF walls.

The tailings will be placed on a 2 mm high density polyethylene (HDPE) lined facility with suitably constructed underdrainage systems. Despite the low acid generating potential and also the presence of carbonate rock, there will be a component of plant waste containing residual thorium and radio-active elements, which will require safe disposal in the TSF.

A preliminary site was chosen for a storage capacity of 13.3 Mt, with a full-containment facility in line with the Global Industry Standards for Tailings Management (GISTM). The design also took cognisance of the potential seismic nature of the area with the full waste containment and has 1V:3H outer perimeter waste rock side-slopes.

The capital expenditure estimate (capex) was factorised from a database of costs into 2023 prices. The overall TSF has an estimated capex of $54 million over the 18-year mine life. It may be possible to divide the capex over various design phases with further design work, to reduce the initial capital and to increase the sustaining capital over the subsequent tailings dam lifts.

1.7.2 Waste rock disposal

The waste rock excavated from the Monte Mumbe open pit mining activities will be loaded and hauled to a permanent disposal site or waste rock dump (WRD). The waste rock will partly be used in the construction of the TSF containment embankments. The balance of the waste will be deposited in designated waste rock dumping between the Target 1 and Target 4 pits. The TSF embankment will require approximately 3.6 Mm^3 or 6.5 Mt of waste rock, with the WRDs requiring a collective capacity of 8.7 Mm^3 or 15.7 Mt.

The WRD is expected to be non-acid forming with limited release of contaminants over the long term. The waste rock contact water is however expected to be high alkalinity (due to the carbonatites) with minor concentrations of REEs. The WRD footprints are not expected to be lined, but an engineered (compacted) basal area is proposed to rescue seepage and protect ground- and surface water resources.

The WRDs will be located adjacent to each respective pit, and adjacent to the TSF. The development of the waste rock dumps will be in 10 m vertical lifts, with 15 m wide benches and 1V:1.5H intermediate side slopes. The overall outer side slope profile will be 1V:3H for rehabilitation. The WRDs will cover a total footprint (natural ground) of approximately 40 ha, with a final downstream height of approximately 50 m.

The capex for the WRDs has been determined through the factorisation of database costs into Y2023 terms and has been estimated at $2.72 million, inclusive of 30% preliminary and general costs. The capex is primarily comprised site clearance and earthworks with selected concrete works and drainage material.

Future recommended work includes a geotechnical investigation and geochemical characterisation of the waste rock.

1.8 Social and environmental matters

Exploration activities on LPP7573L are carried out under an environmental management plan (EMP) prepared by local environmental consultancy GeoAmbiente Lda. The Company’s activities were subjected to an independent Environmental Audit which was validated by the National Agency for Environmental Quality Control (AQUA) of Tete Province on 24 October 2022.

The Licence is not located in any environmentally protected area.

As part of its Mining Concession application, the Company will prepare an EMP covering the proposed mining operations, and subsequently a Level A environmental impact assessment (EIA).

A Level A EIA covers mining activities carried out on a Mining Concession. These activities require a full EIA, which must be prepared by an environmental specialist licensed by the Ministry of Land and Environment (MITA). The EIA process aims at producing a project-specific environmental licence.

The EIA licensing process involves:

- The preparation and submission to MITA of a set of Terms of Reference (ToR), which must include the timing and procedures for public consultation, a risk and emergency management plan, and an EMP.
- The review of the ToR by MITA and the Minister, Ministry of Mineral Resources and Energy (MIREME).
- If the EIA is approved, MITA issues an Environmental Licence within 10 days from the date of approval. The Environmental Licence is valid for the duration of the Mining Concession but must be reviewed every 5 years.

The holder of a Level A Environmental Licence must also submit an annual environmental management report, with the monitoring process carried out either by the concessionaire or by an independent consultant.

Level A activities also require the provision of an environmental bond to cover rehabilitation activities during the closure of the mine. The bond may take the form of an insurance policy, a bank guarantee, or a deposit in cash in a bank account provided by MIREME. The value of the bond is based on an estimate of the costs of such restoration, which will be calculated during or after the active life of the project. The value of the bond is set by MIREME and reviewed every two years.

1.8.1 Radiation management

The Project ore contains low levels of thorium (Th) and uranium (U). The LOM average concentrations for the bastnaesite ore are 200 ppm Th and 20 ppm U at Target 1 and 330 ppm Th and 7 ppm U at Target 4, which is favourably low compared with other rare earth deposits. The Project's flotation tailings will contain lower levels of radioactivity because thorium and uranium are mostly associated with the rare earth minerals (and hence removed from the tailings).

The mineral concentrate produced from the Project, whilst having an upgraded Th and U content, is expected to have a specific activity well below the trigger point of 10 Bq/g and will therefore not be deemed as Class 7 Dangerous Goods for transportation purposes. Note that this concentrate does not leave the site; it is fed directly to the hydrometallurgical plant.

Radionuclides (Th, U and the decay nuclides) will be removed during the hydrometallurgy refining stage to produce a radionuclides-free MREC.

Altona will develop a comprehensive radiation management plan and undertake regular monitoring and regulatory compliance of radioactivity levels of all activities including exploration, mining, processing and tailings disposal.

1.8.2 Closure and remediation

The intent for closure planning at the Project is that disturbed areas will be rehabilitated and closed in a manner to make them physically safe to humans and animals, geotechnically stable, and geochemically non-polluting/ non-contaminating. It is the Company's intent that a sustainable solution is agreed upon for post-mining land use, without unacceptable liability to stakeholders.

In addition, environmental rehabilitation will be ongoing throughout the LOM. Decommissioning activities are likely to include the following:

- Dismantling of buildings and infrastructures.
- Rehabilitating haul roads and hard stand areas.
- Ensuring access to the void left from open pit mining is restricted.
- Reprofiling slopes and top surfaces of waste rock dumps, stockpiles and TSF to ensure stable landforms.

- Revegetation of previously disturbed areas with indigenous vegetation.

1.9 Project costs and economic analysis

Snowden Optiro has undertaken a real financial model for the Project. The base date for all financial inputs is 1 September 2023. All values reported in this section are real; and all diagrams and tables have been generated from the financial model. ROM material and mineralisation are used interchangeably in this section.

A basis of estimate and exclusions are referenced in Section 11.2 and 11.3 respectively, for capex and opex.

A mine schedule has been undertaken by Datamine and reviewed by Snowden Optiro. Proposed ROM steady state production of 0.75 Mt/a is reported for a mine life of about 20 years. A tail-cut has reduced the Project mine life to 18 years and is referenced accordingly as the LOM in this report. Planned steady-state is reached in Year 1 of mining production.

Long-term CIF (China) metal prices calculated as the average of Adamas Intelligence forecast (Adamas Intelligence, 2023) for the period 2024-2040 low case scenario have been applied as follows:

- Praseodymium oxide price of $148,000/ t
- Neodymium oxide price of $156,000/ t
- Terbium oxide price of $1,937,000/ t
- Dysprosium oxide price of $440,000/ t.

Gross revenues total $3,670 million over LOM. Neodymium and praseodymium comprise the bulk of planned gross revenues (86%) along with dysprosium and terbium (14%); no value has been ascribed to the other 13 REOs, primarily cerium and lanthanum. A payability of 90% on the four primary elements in the sold MREC has been applied.

Net revenues include a State royalty of 3% on gross revenues; and payabilities of 90% on MREC product sold. Total MREC produced is 270.7 kt over LOM or 15.0 kt p/a, with an equivalent contained TREO volume of 148.9 kt over LOM or 8.3 t p/a. Net revenues total $3,193 million over LOM.

The planned LOM opex and unit opex is shown in Table 1.3. Process opex accounts for 74% of total opex over LOM.

Table 1.3 Planned LOM opex for the Project

Opex item	Value ($ M)	Unit cost ($/t ROM)	Unit cost ($/t MREC)
Mining	152	11.3	563.0
Process	1,127	83.7	4,164.3
Overheads/ shared services	160	11.9	591.0
Off-mine	80	5.9	294.3
Total	**1,519**	**112.8**	**5,612.6**

Note: MREC – Final Mixed Rare Earth Carbonate product

The total initial and sustaining capital for the Project was estimated to be $339.3 million, which includes project execution;, engineering, procurement construction management (EPCM), contingency and sustaining capital costs. Initial capital is estimated to be $276.3 million and includes all capex over the period October 2023 to December 2028. The initial capital is summarised in Table 1.4.

Table 1.4 Initial capital summary

Initial capital item	Value ($ M)
Project mobilisation and camp construction	4.0
Bulk and other infrastructure	31.3
Direct plant costs	150.0
Indirect plant and EPCM costs	35.0
Tailings dam	18.0
Waste rock dump	2.0
Mining infrastructure, pre-production and mobilisation	14.0
Exploration, evaluation, Owners Team and sterilisation drilling	22.0
Total initial capital	**276.3**

Note: EPCM – Engineering, procurement, construction management; rounding has been applied to select initial capital items.

For the LOM, debtors days of 30 days has been applied, creditors of 30 days (mining and process opex) and 15 days on inventories (select mining and process opex).

A production tax or royalty is payable based on the value of the mineral extracted, with an applicable royalty of 3% for other minerals. Total State royalties over LOM is $110.1 million and have been included under net revenues.

A corporate tax of 32% on cashflows (after the applied WPT) has been applied in the financial model. Total corporate tax over LOM is $372.5 million.

Provision has been made under Owners costs, for customs and duties; although there is a strong likelihood that no customs will be payable during the initial years of construction, ramp-up and first two years of steady-state production.

No government free carry has been applied to the financial model.

No capital gains, withholding or transaction tax has been applied.

Snowden Optiro is not aware of any municipal fees or rates that are to be applied.

1.9.1 Net present value (NPV) and internal rate of return (IRR)

The NPV of the Project is $283.3 million, based on a real discount rate of 8%. An NPV of $149.6 million is reported using a real discount rate of 12%. A post-tax IRR of 25% and a payback from the construction start date of 4.5 years, and a payback from first TREO production of 2.5 years is reported. An operating cashflow margin of 42% is noted. Project earnings before interest, tax, depreciation and amortisation (EBITDA) would effectively be operating cash flows (no capital expenditure, tax, interest, depreciation nor amortisation expenses have been included). Operating cashflows would include all realisation costs, on- and off-mine expenses and royalties. The planned LOM EBITDA will be $1,674 million; and planned annual EBITDA is $93 million.

1.9.2 Sensitivity analysis

Using an NPV of $283.3 million with an applied real discount rate of 8%, the Project is most sensitive to revenue (price, recovery, grade and exchange rates), less sensitive to opex and least sensitive to capex (Figure 11.6). The sensitivity analysis shows that the Project is more sensitivity to capital than other benchmarked projects.

1.9.3 Summary of key Project parameters

A summary of key Project parameters is shown in Table 1.6.

Table 1.5 Forecast key Project parameters

Parameter	Unit	Value
Ore processed	Mt	13.5
TREO ROM grade (after dilution)	%	2.30%
MREC produced	Kt	270.7
Initial capex	$ M	276.3
Sustaining capex	$ M	63.0
Opex LOM	$ M	1,519.3
Opex per sold MREC	$/t MREC	5,612.6
Gross revenue LOM	$ M	3,670.2
Net revenue LOM	$ M	3,193.1
EBITDA LOM	$ M	1,673.8
Gross revenue per tonne MREC	$/t	13,558.4
Net revenue per tonne MREC	$/t	11,795.8
Payback from first MREC	Years	2.5
Post-tax NPV_8	$ M	283.3
Post-tax NPV_{10}	$ M	207.0
Post-tax IRR	%	25%
Operating margin	%	42%

Note: TREO – Total Rare Earth Oxide; ROM – Run of mine; MREC – Mixed Rare Earth Carbonate; EBITDA – Earnings before interest tax, depreciation and amortisation; opex – operating expenditure.

1.9.4 Upside scenario

An upside scenario with higher long-term metal prices has been undertaken. No changes in production, opex, capex or discount rates were made to the financial model. The long-term metal prices applied are as follows:

- Praseodymium oxide price of $174,000/ t.
- Neodymium oxide price of $183,000/ t.
- Terbium oxide price of $2,083,000/ t.
- Dysprosium oxide price of $474,000/ t.

Total gross revenues of $4,258 million are reported over LOM for the upside scenario; with planned net revenues of $3,704 million.

The NPV of the upside scenario is $409.9 million, based on a real discount rate of 8%. An NPV of $231.3 million is reported using a real discount rate of 12%. A post-tax IRR of 32% and a payback from the construction start date of 4.0 years, and a payback from first TREO production of 2.0 years is reported. An operating cashflow margin of 50% is noted.

1.10 Project execution

A high-level planned schedule has been undertaken for the overall Project. Key milestones are highlighted in the Level 1 schedule (Table 1.6). The schedule was based on industry benchmarking, scope of work and a general deliverables list. Snowden Optiro assumes a seamless advancement between the various phases, as the Project advances. The overall schedule is five years to first TREO being produced, which includes 18 months for a PFS, one year for a FS, two years construction and a six-month production ramp-up. Project financing will be applied for, for pre-production funding and Project construction.

An engineering, procurement, construction management (EPCM) execution strategy has been recommended for the Project.

Table 1.6 Planned milestones for the Project

Milestone	Milestone date/ duration
Submission of Mining Concession application	Q4 2023 (achieved)
Prefeasibility study	18 months to March 2025
Feasibility study	12 months to March 2026
Value engineering, FEED and financing	Nine months to December 2026
EPCM tendering	November 2026
Early works commencement	December 2026
EPCM award	January 2027
Construction commences	Two years to December 2028
First TREO to be produced	December 2028
Production ramp-up	Six months to June 2029
Steady state of 187.5 kt per quarter (750 kt/a)	Q3 2029

Note: FEED – Front end engineering design; EPCM – Engineering, procurement, construction management; TREO – Total Rare Earth Oxide

Source: Snowden Optiro, 2023

1.11 Recommendations

It is expected that Altona will undertake a prefeasibility study (PFS) as the next stage of project development, based on the positive outcome of this CPR and Scoping Study.

1.11.1 Exploration

Snowden Optiro's recommendations for continued exploration include:

- Use the improved mineralisation model to attempt identifying new targets, including blind targets.
- Continue improving mineralisation model through mapping as well as academic research.
- Exploration drilling at T3, T9, T11, and any other potential high-grade target
- MRE update
- Data centralisation

The resource update should cover tonnage increase, as well as improve the level of confidence within the pits to Measured and Indicated.

1.11.2 Geometallurgy / processing

Geometallurgy and process flowsheet design will be a priority activity during the PFS. The Scoping Study sighter testwork forming part of this Scoping Study provides a preliminary assessment based on a possible flowsheet. Ongoing work includes:

- Mineralogical and geo-metallurgical assessment.
- Beneficiation flowsheet development.
- Hydrometallurgical flowsheet development.

Further detailed metallurgical studies for Monte Muambe are underway and currently focused on advanced metallurgical testwork. A 70 kg representative ore sample is with Auralia Metallurgy in Perth (Australia), and another 100 kg ore sample has been received by SGS Lakefields in Canada. The sample at SGS Lakefields will first undergo extensive feed characterisation including electron microprobe analysis and TIMA-X analysis. TIMA-X is designed to provide quantitative mineral speciation and distribution, as well as characterisation, grain size attributes, degree of liberation and associations of minerals of interest.

Following feed characterisation, test work will focus on producing a high-grade Rare Earth concentrate in order to improve the economics of the Mixed Rare Earth Carbonate production process.

1.11.3 Mining

Snowden Optiro’s recommendations include:

- Geotechnical studies:
  - Drilling program
  - Geotechnical logging of core
  - Off-site testing of core
  - Structural interpretation
  - Slope stability assessment
- Mine planning and ore reserve:
  - Pit optimisation and schedule
  - Scenario analyses
  - Cost assessments
  - Ore reserve development

Once adequate testwork has been completed to reliably inform geotechnical models, recovery and process cost parameters, more detailed work can be carried out.

1.11.4 Environmental studies

This will involve starting baseline studies as soon as possible and planning to reach environmental compliance as part of the Mining Concession application (EMP, ESIA). This will include environmental, social and governance (ESG) planning to a World Bank level. Minimisation of the carbon footprint of the proposed product can be minimised through locally available sourcing.

1.11.5 Infrastructure studies

This will involve multiple trade-off studies; including logistics optimisation (road vs different rail options), and power sources mix optimisation (based on capex, opex and carbon footprint).

1.11.6 Tailings / waste management

The PFS study will require a detailed site selection and associated surface geotechnical investigations. A key requirement will be to conduct geochemical static and kinetic leach testing on the types of ore / tailings / waste to determine the future design / lining of the TSF. The planned PFS will identify several options and determine the best site or sites for tailings disposal.

For waste rock disposal, the prefeasibility scope will include hydrogeological testing, site selection, geotechnical and chemical testwork. Both tailings and waste rock will require detailed design criteria and opex / capex costings.

1.11.7 Marketing

As part of the PFS, Altona will join a Responsible Sourcing organisation and integrate Responsible Sourcing processes. The Company plans to develop marketing side of business as part of PFS, which may include offtakes and integration with rest of world supply chains (existing and projects).

1.11.8 Project economics

Relevant studies need to be undertaken to improve granularity and accuracy of the opex and capex estimates, production, payabilities and planned recoveries.

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 Background

This Competent Person's Report (CPR) and Scoping Study document has been prepared by Snowden Optiro, a business unit of Datamine Australia Pty Ltd (Snowden Optiro) for Altona Rare Earths Plc (Altona, the Company or the Client). This CPR has been prepared to provide compliant disclosure on all material mining assets and liabilities of Altona's Monte Muambe Rare Earth Element (REE) project (Monte Muambe or the Project), located in western Mozambique, in accordance with relevant Financial Conduct Authority (FCA) guidelines (FCA Technical Note 619.1 of May 2022, Appendix II), and the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 edition (JORC Code, 2012).

2.2 Reporting currency

In this CPR and Scoping Study all references to dollars ($) refer to American United States dollars.

2.3 Principal sources of information

In the compilation of this CPR and Scoping Study, Snowden Optiro used data and documents as provided by Altona via dataroom and direct file transfer. Mr Cedric Simonet, CEO of Altona was the primary contact for data. The primary information sources were the drilling, survey and exploration data provided by Altona, along with studies, internal reports and correspondence prepared by the Altona team.

2.4 Site visits

A site visit was conducted by the Competent Person's representative, R.N. Barnett from 7 to 10 August 2023 (two days at Project site). RC drilling and sampling procedures were inspected, check tests conducted on core density samples and a project review conducted using a checklist.

2.5 Effective date

The effective date of this report is 24 April 2024. The authors are not aware of any material change in the status of Altona's Project in the period between receipt of data and completion of the CPR.

2.6 Independence

At the date of this CPR and Scoping Study, Snowden Optiro had no association with Altona or its individual employees, or any interest in the securities of Altona or any other interests that could reasonably be regarded as capable of affecting its ability to give an independent unbiased opinion in relation to Altona's assets.

Snowden Optiro will be paid a fee for the preparation of this CPR and Scoping Study based on a standard schedule of rates for professional services, plus any expenses incurred. This fee is not contingent on the outcome of the CPR or Scoping Study, and Snowden Optiro will receive no other benefit for the preparation of this report.

2.7 Qualifications of the consultants and Competent Persons

The Competent Person and principal author responsible for preparation of this of this CPR and Scoping Study is Mr Julian Aldridge of Snowden Optiro.

The contributions and responsibilities of each of the co-authors to this CPR are detailed in Table 2.1

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Table 2.1 Responsibilities of each co-author

Author	Qualifications	Responsible for section/s
Dr Andrew Scogings	*PhD Geology, MAIG, RPGeo (Industrial Minerals)*	Competent Person (MRE), Section 4 Geology and Mineral Resources
Mr Kahan Cervoj	*BAppSci (Geology), Post Graduate Certificate (Geostatistics), MAusIMM, MAIG*	Section 4 Geology and Mineral Resources
Mr Robert Barnett	*BSc Eng. (Mining Geology) MSc Industrial Mineralogy*	Section 4 Geology and Mineral Resources
Mr Vince Agnello	*BSc (Hons) Geology, GDE, M Eng (Min Econ), MSAIMM, MGSSA, Pr Sci Nat*	Section 11 Costs and economic evaluation, Section 12 Project execution
Mr Julian Aldridge	*MSc Mining Geology (MCSM), MESci (Oxon), CGeol FGS, MIMMM*	Overall Scoping Study CP and project management
Mrs Beatriz Zanoli Sato	*BSc Mining Engineering*	Section 5 Mining
Mr Allan Earl	*FAusIMM, WASM*	Section 5 Mining
Mr Peter Theron	*BEng (Civil), GDE, MSAIMM, PrEng*	Section 8 Tailings and waste management
Mr Stephan Geyer	*BEng (Civil)*	
Mr James Norton	*BSc(Eng)(Civil), GDE(Civil), MEng*	Section 7 Project infrastructure
Mr Gavin Beer	*BSc (Metallurgy), MAusIMM (CP)*	Section 6 Metallurgy and processing

2.8 Reliance

Snowden Optiro is responsible for this CPR document and included Scoping Study. Snowden Optiro declares that it has taken all reasonable care to ensure that the information contained in this report is, to the best of its knowledge, in accordance with the facts and contains no material omissions.

In preparing the contained MRE, Snowden Optiro and the authors have relied on information collated by other parties. Snowden Optiro and the co-authors have critically examined this information, made their own enquiries, and applied their general mineral industry competence to conclude that the information presented in this MRE is done in accordance with the definitions and guidelines of the JORC Code (2012).

Snowden Optiro insists that its opinions must be considered as a whole, and that selection of portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in this Scoping Study and CPR. The preparation of a Scoping Study and CPR is a complex process and does not lend itself to partial analysis or summary.

2.9 Limitations

Altona has confirmed in writing to Snowden Optiro that, to its knowledge, the information provided by it (when provided) was complete and not incorrect or misleading in any material respect. Altona has agreed to indemnify Snowden Optiro from any liability arising as a result of or in connection to the information provided by or on behalf of Altona being incomplete, incorrect or misleading in any material respect.

3 PROJECT DESCRIPTION AND LOCATION

The Project is an advanced exploration and pre-development project for which a Scoping Study and CPR have been prepared by Snowden Optiro in October 2023. Altona's interest in the Prospecting Licence (Licença de Prospecção e Pesquisa) LPP7573L (the Licence) is through a Farm Out Agreement dated 23 June 2021 between Ussokoti Investimentos, Altona, Monte Muambe Mining Lda (MMML) and its original shareholders. The Farm Out Agreement gives Altona the right to earn up to 70% of MMML in a phased manner, subject to the completion of certain conditions and milestones.

The Project is a proposed greenfield operation. The direct environmental liabilities of Altona are therefore limited to the closure and rehabilitation of previous and current exploration sites as required by an Environmental Management Plan (EMP) that must comply with the provisions of the Environmental Law (Law no 20/1997 of 1 October), the Mining Law (Law no 20/2014 of 18 August 2014), and the Environmental Regulations for Mining Activities (Decree no 26/2004 of 20 August 2004).

3.1 Project location and Infrastructure

The Project is located in Niculunga Locality, Cambulassisse Administrative Post, Moatize District, Tete Province, Mozambique (Figure 3.1). Access from the provincial capital Tete is through the tarred road leading to the Malawi border (Zobue), through Moatize town, and to the village of Cateme. The distance from the Zambezi bridge in Tete to the Cateme turn-off is about 40 km.

From the Cateme turn-off, an ungraded track leads from to the Monte Muambe camp through the villages of Mualadzi and Djendje. The distance is approximately 43 km. Access can become difficult during severe rain episodes.

Figure 3.1 shows the Project location in terms of primary infrastructure.

The shortest road from Tete to Moatize is currently not usable due to the collapse of the Rio Revuboe bridge in 2022 as a result of cyclone Ana. The alternative route involves crossing the Zambezi through the Kassuende bridge, located about 5 km southeast of Tete, and joining the Moatize road through a road passing south of the Rio Revuboe. An international airport is available in Tete, with twice-daily flights to Maputo and four flights a week to Johannesburg.

Tete is a logistics hub connecting Mozambique to Malawi, Zambia and Zimbabwe, as well as to the Indian Ocean. Two railway lines connect Moatize to the Indian Ocean (Figure 3.1), namely the Sena line to the port of Beira (560 km) and the Nacala Corridor to port of Nacala (910 km). The Sena line passes about 20 km to the NW of the Project site. The distance from the Project site to the nearest railway siding in Moatize is about 65 km.

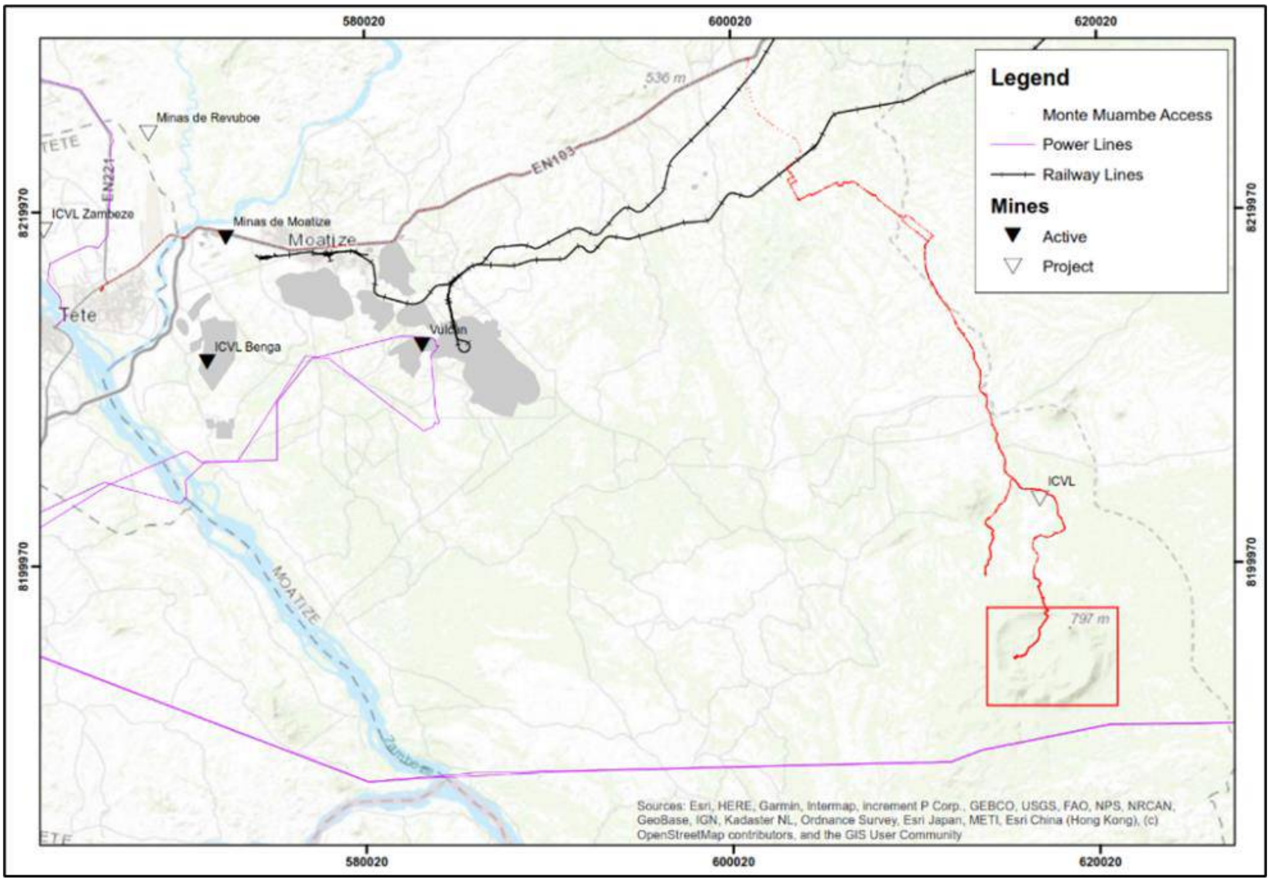
The Tete Province is host to several large coal mines (Jindal, ICVL and Vulcan), a major iron and steel project (Baobab Steel), as well as several coal and coal-based power generation projects (Figure 3.2). In 2016 the Mozambique government set up the Revuboe Industrial Free Zone in Chiuta District, near the Baobab Steel project location, on an area covering about 4,800 ha.

The Project is located about 165 km from the 1,450 MW Cahora Bassa hydroelectric plant. In December 2022, the company operating the plant secured a $125 million loan to rehabilitate and modernize the plant. The renovations are expected to take the plant's power generating capacity to 2,075 MW and to be concluded in 2025.

Access to cell phone networks at Monte Muambe is presently limited to the Camp area (Vodacom and Movitel operators). The Company is in discussions with Vodacom to instal a repeater on site, which will allow good cell phone coverage in the entire Project area, and which will also benefit neighbouring communities. Internet access for the camp is presently provided through a 3 mbps microwave connection.

Exploration boreholes have encountered water in several areas of the Project site and water for the camp is presently drawn from one of these boreholes.

Figure 3.1 Project infrastructure map



3.2 Physical environment

Monte Muambe forms a circular ridge, with a diameter varying from 4.8 to 5.4 km. While the structure resembles a volcanic crater, the ridge consists of indurated sandstones and the structure is a differential erosion feature (Figure 3.2).

The elevation of the top of the ridge ranges from 600 m to 735 m above mean sea level (amsl), with the surrounding plain having an elevation of 400 m to 425 m amsl. On the southern side, the ridge is incised by two narrow valleys which act as drainages for the inner part of the structure. Drainage is in a southerly direction, towards the Zambezi gorge, about 35 km to the south (Figure 3.3).

The inner part of the basin is about 3.5 km in diameter and forms a relatively level surface oscillating between 550 and 580 m amsl, with isolated carbonatite hills. Carbonatites outcrops have undergone karstification, with caves visible at the surface, and cavities also encountered in some of the drill holes.

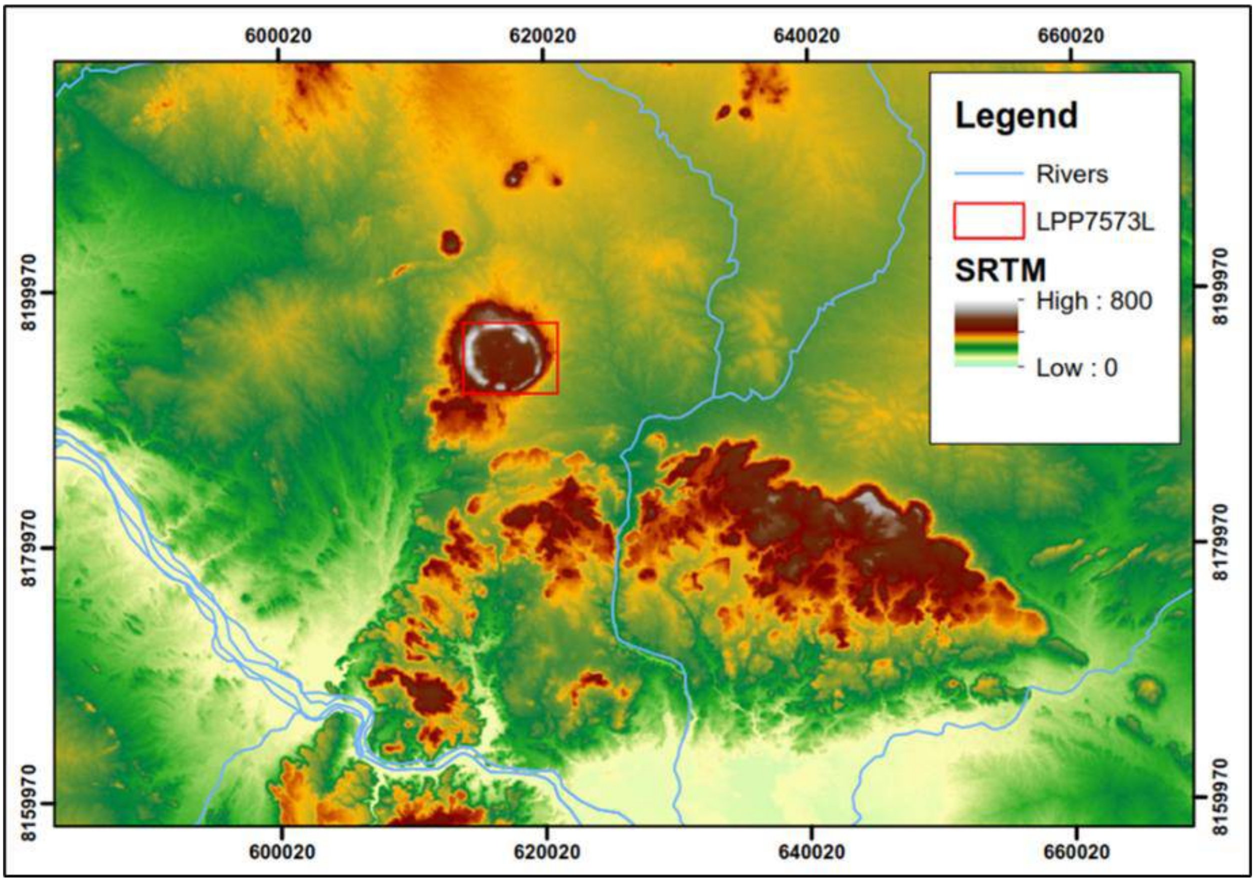
The outer part of the ridge shows a step profile corresponding to paleosurfaces at 550 m, 610 m and 630 m amsl in particular. Soil in the basin and on the ridge slopes are thin, typically less than 1 m.

Beside the Company's camp, there are no human settlements within the Licence. Human activity within the Licence area is limited to neighbouring community honey harvesting, logging and hunting.

There are five villages located around the Licence, namely Djendje (1 and 2), Chincolo, Cachenga 1 and 2. The majority of the Company's employees come from these villages. The population density around the Project area is 15 people per km^2 in average. Vegetation on the slopes of the ridges consists of a dense forest. In the basin, the vegetation density varies based on the topography.

Figure 3.2 **Aerial view of Monte Muambe, looking south**

Figure 3.3 **SRTM DTM of the Lupata area and Monte Muambe**

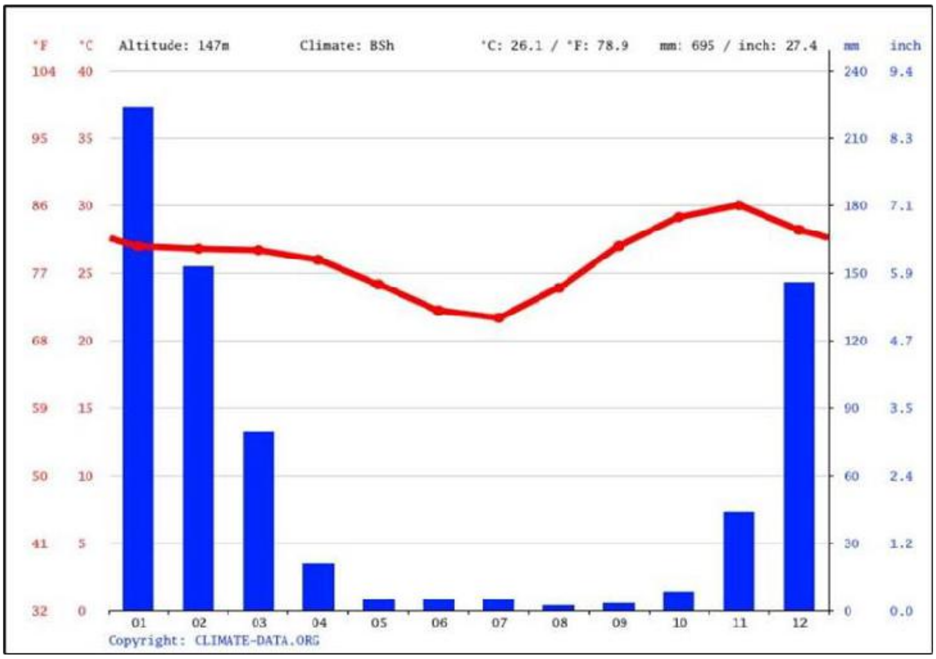


3.3 Climate

The nearest specific climatic data obtainable is for Tete. In general, temperatures at the crater rim can be a few degrees cooler than in Tete (which is far lower in elevation), with an elevation difference of up to 600 m on the crater rim and 400 m inside the crater. The climate of Tete is tropical semi-arid, with a hot, rainy and mostly cloudy period from December to March (rainy season) and a long dry windy and mostly clear season from April to October (dry season), within which there is a relatively cool period from June to August. Tete, being located in central-western Mozambique at a low altitude along the Zambezi River, is the hottest area of the country (Figure 3.4). The most intense heat waves occur in the last months of the year, before the rainy season. Over the course of the year, the temperature typically varies from 18°C to 36°C and is rarely below 16°C or above 41°C.

From December to February rains can affect exploration activities (and planned mining operations), and access conditions can become difficult during rainy episodes; however, the Company has so far been able to carry out activities during the months of February and March.

Figure 3.4 Tete annual climate data



3.4 Ownership structure

Altona's interest in the Licence (Table 3.1) is through a Farm Out Agreement dated 23 June 2021 between Ussokoti Investimentos (the original owner of the licence), Altona, MMML and its original shareholders. The Farm Out Agreement gives Altona the right to earn up to 70% of MMML in a phased manner, subject to the completion of certain conditions and milestones. Each transfer of shares requires the approval of the Minister, Ministry of Mineral Resources and Energy (MIREME).

Altona currently has a holding of 51% in MMML after receiving formal regulatory approval from the Mozambique Minister of Mineral Resources and Energy on 15 January 2024.

Table 3.1 Summary of the Monte Muambe Farm Out Agreement of 23 June 2021

Phase	Conditions for completion of each phase**	Altona holding in MMML on completion of each phase
Phase 1 (8 months, completed)	GBP40,000 in cash 1 million Altona shares Minimum Expenditure $400,000	20%
Phase 2 (12 months, completed upon submission of MRE and Scoping Study on 18 October 2023).	GBP40,000 in cash 1 million Altona shares Minimum expenditure $700,000 Production of a JORC Code (2012) MRE & Scoping Study	51%
Phase 3 (24 months)	GBP160,000 in cash (in instalments) 1 million Altona shares Minimum expenditure $2 million Production of a feasibility study	70%

*Note: ** Payments in cash and shares are to the original shareholders of Monte Muambe Mining Lda. Altona – Altona Rare Earths Plc; MMML - Monte Muambe Mining Lda*

Source: Altona, 2023

Post Phase 3 the original shareholders of MMML will remain with a 20% free carried interest and a 10% participating interest. Altona can buy all or part of original shareholders' participation. Altona controls MMML through the appointment of two out of three Directors (as per the Farm Out agreement), and the appointment of the Managing Director.

3.5 Project tenure

The Project is held under the Licence issued in accordance with the Mining Law 2014 by the Ministry of Mineral Resources and Energy (MIREME). The Licence LPP7573L covers a surface area of 3,940 Ha (39.40 km^2), and is valid for fluorspar, rare earths, and associated minerals (Table 3.2 and Figure 3.5). The Licence was granted to Ussokoti Investimentos Sociedade Unipessoal for an initial five year term, and was issued for the period 22 May 2017 to 22 May 2022. Ussokoti requested an extension of the period of the prospecting licence and a transfer of LPP7573L to MMML. MMML is a Special Purpose Vehicle (SPV), setup for purposes of Altona's earn-in into the Project. In terms of Mozambican laws, a prospecting licence may be issued for an initial period of five years and renewed for an additional three year period.

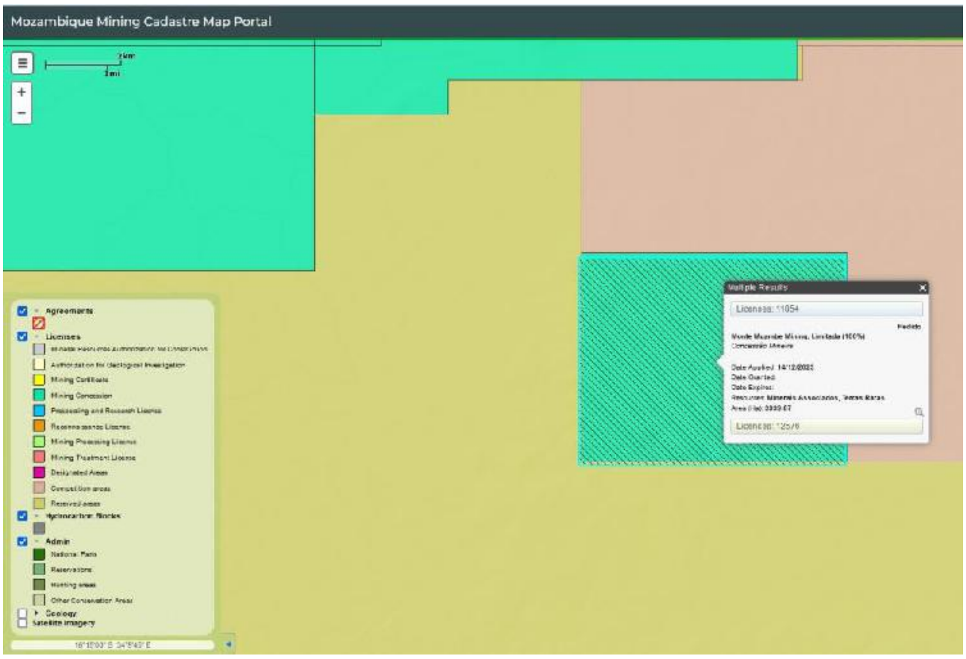
On 26 October 2022, INAMI notified MMML that the licence had been transferred to it from Ussokoti and renewed for a further three-year term, up to and including 22 May 2025. As at the date of this report, the Mozambique Mining Cadastre Portal indicates that the Licence LPP7573L is held by MMML, and expires on 22 May 2025 (MMCMP, 2023).

MMML has lodged an application for a Mining Concession (Mining Licence). The application follows the successful completion of Phase 2 of the Project. The requested duration for the Mining Concession is 25 years. The Mining Regulations (Decree no. 31/2015 of 31 December) require the Minister of Mineral Resources and Energy to communicate their decision to the applicant within 190 days.

Table 3.2 Mining Concession corners for LPP7573L as granted.

Point	Latitude		Longitude	
	Degrees	Minutes	Degrees	Minutes
1	-16	18	34	4
2	-16	18	34	8
3	-16	18	34	8
4	-16	18	34	4

Figure 3.5 Screenshot from the Mozambique Mining Cadastre map



3.6 Altona commitments

As of 15 January 2024, MMML is held 51% by Altona. In terms of the agreements between Altona, Ussokoti and MMML, Altona will bear 100% of the Project costs up to completion of Phase 3 and holds a majority position on the board of MMML. The earn-in arrangement between the companies is broadly as shown in Table 3.1.

Beside exploration rights, the Prospecting Licence gives the licensee a preferential right to an application for a Mining Concession. The Company lodged a Mining Concession (Mining Licence) application on 14 December 2023.

Mining Concessions have a validity of up to 25 years, renewable once for an equal period. Mining activities on a Mining Concession also require the obtention of Land Rights (Direito de Uso e Aproveitamento da Terra – DUAT), and an Environmental Impact Study for Category A activities.

The work and minimum expenditure commitments are as follows:

- Phase 1 (8 months): 3,000 m exploration drilling programme, with a minimum expenditure commitment of $400,000. This is complete.
- Phase 2 (12 months): In-fill drilling programme to produce a maiden Mineral Resource Estimate to establish the Total Rare Earths Oxide (TREO) present, and first pass metallurgy which is a key parameter for REE projects, with a minimum expenditure commitment of $700,000. This is complete.
- Phase 3 (2 years): Preparation of a feasibility study (FS), with a minimum expenditure commitment of $2 million. The Mining Concession application of 14 December 2023 forms part of the FS objective.

3.7 Servitudes

There are no known servitudes (communication, water, road), current or required, within the Project area.

3.8 Mozambique mineral law

The current law came into force on 18 August 2014 (Norton Rose Fulbright, 2014). Licences can be awarded to any legal person established and registered in Mozambique who have the required technical and financial capacity. The following licences are available (Thomson Reuters, 2020):

- Prospecting and Research Licence (Licença de Prospecção e Pesquisa or LPP), ◦ Up to a maximum area of 19,998 ha for non-construction minerals (such as REE) and issued for a period of up to 8 years.
- Mining Concession (Concessão Mineira). Valid for a period of 25 years and can be extended for a further period of 25 years and confers the right to extract, develop and process mineral resources discovered under an LPP.
- Mining Certificate (Certificado Mineiro), relevant mainly to small-scale artisanal mining activities. Granted to Mozambican nationals and legal entities.
- Mining Pass (Senha Licença). Relevant mainly to small-scale artisanal mining activities. Granted to Mozambican nationals and legal entities.
- Mining Treatment Licence (Licença de Tratamento Mineiro).
- Mining Processing Licence (Licença de Processamento Mineiro).
- Licence for the Commercialisation of Mining Products (Licença de Comercialização de Produtos Mineiros). Governs the activity of the sale and purchase of mineral products sourced from outside of Mozambique.

Following from a prospecting licence (LPP), large scale mining must be carried out under a Mining Concession.

Royalties and taxes which will become applicable in the instance of any mining include:

- Income tax.
- Value added tax (VAT).

- Production tax (in essence a royalty of 3% in the case of REE).
- Surface tax (related to area held).
- Municipal taxes.
- Any other taxes required by law.

Mining activities require a full environmental impact assessment (EIA) and the mining company must provide a bond to cover the costs of environmental restoration during the closure of the mine. The bond can be an insurance policy, a bank guarantee or a deposit in cash in a bank account provided by MIREME. The amount of the bond is based on an estimate of the costs of the restoration (calculated during or after the active life of the project). The amount is set by MIREME and is reviewed every two years. For mining, the amount is based on the terms of the EIA.

3.9 Surface rights

Under Mozambican law, the land is property of the State. Investors in mining activities cannot, therefore, buy or own land being used for the implementation of a mining project. Mining investors may be granted the right to use and exploit the land, known as Direito do Uso e Aproveitamento da Terra (DUAT).

A DUAT provides its holder with legal certainty that it will be authorised to use a certain area of land for the purposes for which the DUAT was granted, such as mining activities. DUAT holders may also be owners of buildings, facilities or other immovable assets built on the land covered by their DUAT. When mining rights are awarded over an area of land subject to an existing third-party DUAT, the holder of the mining rights must pay compensation to the respective DUAT holder.

In cases where a mining right is awarded over a populated area and the population must be resettled, a relocation plan must be drawn up and due compensation paid. The DUAT application will be made once the Mining Concession application has been submitted.

3.10 History

One of the first references on the geology of Monte Muambe is the February 1930 issue of the Geological Magazine (Dixey, 1929). Dixey describes Monte Muambe as consisting of high ridges of hard grits enclosing crystalline limestone rich in iron ores.

Initial exploration work at Monte Muambe took place in the early 1960s and is reported by Dias (1961). Work done by the Bulgargeomin brigade in 1983 focused on fluorite occurrences and is reported in Cilek (1986).

Grupo Madal carried out exploration for fluorspar at Monte Muambe in 1998. This included a helicopter borne magnetometer and gamma spectrometer survey, flown mean terrain clearance of 35 m and 60 m respectively, with a line spacing of 100 m and a tie line spacing of 500 m. Several trenches were also dug on outcropping fluorspar mineralisation. According to Siegfried (2021), a fluorite bulk sample was also collected and submitted to South African metallurgical company Mintek for beneficiation test work.

In 2009 Globe Metals & Mining Ltd, an ASX-listed company, entered into an agreement with the then owner of the licence (Bala Ussokoti) under which Monte Muambe was held at the time to acquire and explore the project. Exploration work was initially focused on fluorspar. On 9 March 2012, Globe Metals & Mining published a maiden Inferred fluorspar Mineral Resource totalling 1.63 mt at 19% fluorite, for a total of 310,000 t of fluorite contained. Globe Metals & Mining discovered REE occurrences during fluorspar exploration, initially in rock chip samples, and in 2011 in reverse circulation (RC) boreholes. On 14 March 2011, Globe Metals & Mining announced multiple REE discoveries at Monte Muambe over 3 distinct zones (AA, BB, and DD) of the Monte Muambe basin.

Intercepts reported at the time included:

- 46 m at 2.6% TREO including 20 m at 3.3% TREO from 24 m (Zone AA).
- 49 m at 2.5% TREO including 20 m at 3.5% TREO from 20 m (Zone BB).
- 60 m at 2.1% TREO including 24 m at 2.6% TREO from 20 m (Zone DD).

Further drilling in 2012 continued to yield REE intercepts of up to 96 m at 2.2% TREO. Globe Metals & Mining drilled 165 RC holes for a total of 12,587 m.

While Globe Metals & Mining announced plans to carry out metallurgical testing, no such test results were reported. In its June 2013 annual report Globe Metals & Mining announced that after spending Australia $5.1 million, it was impairing the mineral asset due to an ongoing legal dispute with the JV partner over the legal right to the tenement.

On 22 May 2017, Ussokoti Investimentos Sociedade Unipessoal, a Mozambican company, secured the Licence, and carried out limited exploration for rare earths and guano. On 23 June 2021, Ussokoti Investimentos Sociedade Unipessoal and Altona entered into the Farm Out Agreement described in section 3.3 of this report.

Rare earths exploration resumed in August 2021. Since then, Altona has invested about GBP£1.85 million in exploration. This includes:

- 5 diamond drill hole (DD) holes (total 590.7 m) drilled in 2021.
- 82 RC holes (total 6,678.8 m) drilled over 3 phases in 2021, 2022 and 2023.
- 26 short (25 m deep) (total 593.83 m) RC holes drilled in 2022.
- 2,960 samples assayed by Intertek Genalysis in Perth for major and trace elements including REE and fluoride.
- A soil sampling survey over the entire inner part of the Monte Muambe basin, for a total of 2,146 samples which were assayed using the Company's portable XRF (pXRF) on site.
- Mineralogical and metallurgical testing.

4 GEOLOGY AND MINERAL RESOURCES

4.1 Regional geology

The geology of Tete Province was extensively mapped as part of the Mineral Resource Management Capacity Building Project (MRMP) between 2002 and 2006 by a consortium led by the Geological Survey of Finland (Geologian TutKimuskeskus, abbreviated as GTK), for the Ministry of Mineral Resources. The geology of the area around Monte Muambe is described in the Tambara Sheet no 1634 (GTK Consortium, 2006) shown in Figure 4.1.

Monte Muambe is located in the central part of the Karoo Moatize-Minjova coal basin, which corresponds to the eastern part of the Zambezi Graben. The basin is separated from the central Sanângoè-Mfidezi Basin by the Cahora Bassa Horst. In the Monte Muambe area, Karoo sediments are bound to the Southwest by a normal fault marking the edge of the Zambezi Graben. To the north, they are bound by the discordance between the Lower Karoo Matinde Formation and the underlying Mesoproterozoic Tete gabbro suite. The Matinde formation is coal bearing, and coal occurrences are known immediately north of Monte Muambe. Phanerozoic formations around Monte Muambe dip gently towards the south.

South of Monte Muambe, the Karoo sediments are covered by the Lupata Group volcanics and sediments, in the following sequence (from base to top):

- Tchazica Formation (sandstones and conglomerates).
- Monte Palamuli formation (rhyolites) and contemporaneous Monte Mazambulo Formation (conglomeratic sandstones).
- Monte Linhanga Formation (phonolites).

The Monte Muambe carbonatite intrusion is hosted by Upper Karoo Sandstones of the Cádzi Formation. The age of the intrusion is presently unknown. It is usually assumed that the intrusion is Cretaceous and forms part of the Chilwa Alkaline Province.

Isolated trachyte intrusions form small hills located 8 to 11 km to the north, and 22 km to the northwest of Monte Muambe. They are considered by GTK Consortium (GTK Consortium, 2006) to be contemporaneous of the Monte Muambe carbonatite. Other Post-Karoo alkaline and carbonatite intrusions in the region include the Salambidue syenite, about 50 km to the north-northeast of Monte Muambe (mostly in Malawi), as well as Cone Negose, located 315 km to the west-northwest of Monte Muambe along the northern margin of the Zambezi Graben.

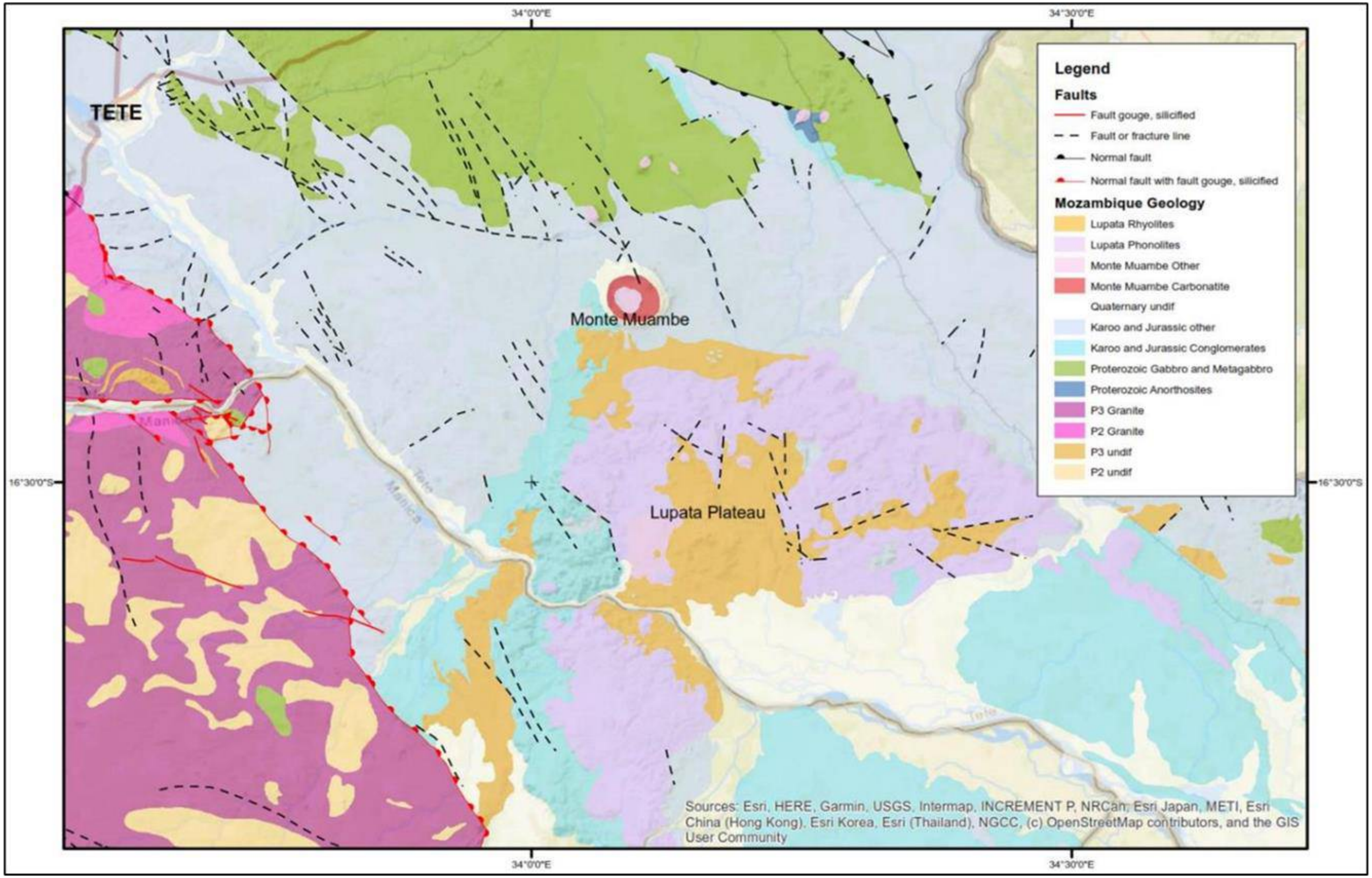
4.2 Local geology

4.2.1 Description

While the Monte Muambe structure resembles a ring-dyke, or a volcanic edifice, the outer ridge actually consists of sub-horizontal indurated Upper Karoo sandstones and is the product of differential erosion (Figure 4.2). The basin formed by the inner part of the structure consists chiefly of fenites, various types of carbonatites, breccias, as well as pyroclastics. The diameter of the carbonatite intrusion at surface level is about 3.3 km. Carbonatites tend to outcrop in the form of small hills rising above the floor of the basin. Fenites are often deeply weathered at near-surface levels and rarely outcrop, though float can be encountered on slopes.

Fenites form a circular zone lining the contact between carbonatites and host sandstones, but the detailed relationships between fenites and carbonatites are a lot more complex, involving faulting during and after the emplacement of the intrusion, as well as the incorporation of xenoliths of various size (centimetre to decimetre size). Drilling in various parts of the intrusion shows that fenite outcrops often cover carbonatites. This, as well as the presence of pyroclastics, suggests that the present erosion level may corresponds to the roof of the carbonatite intrusion, immediately under the base of the volcanic edifice.

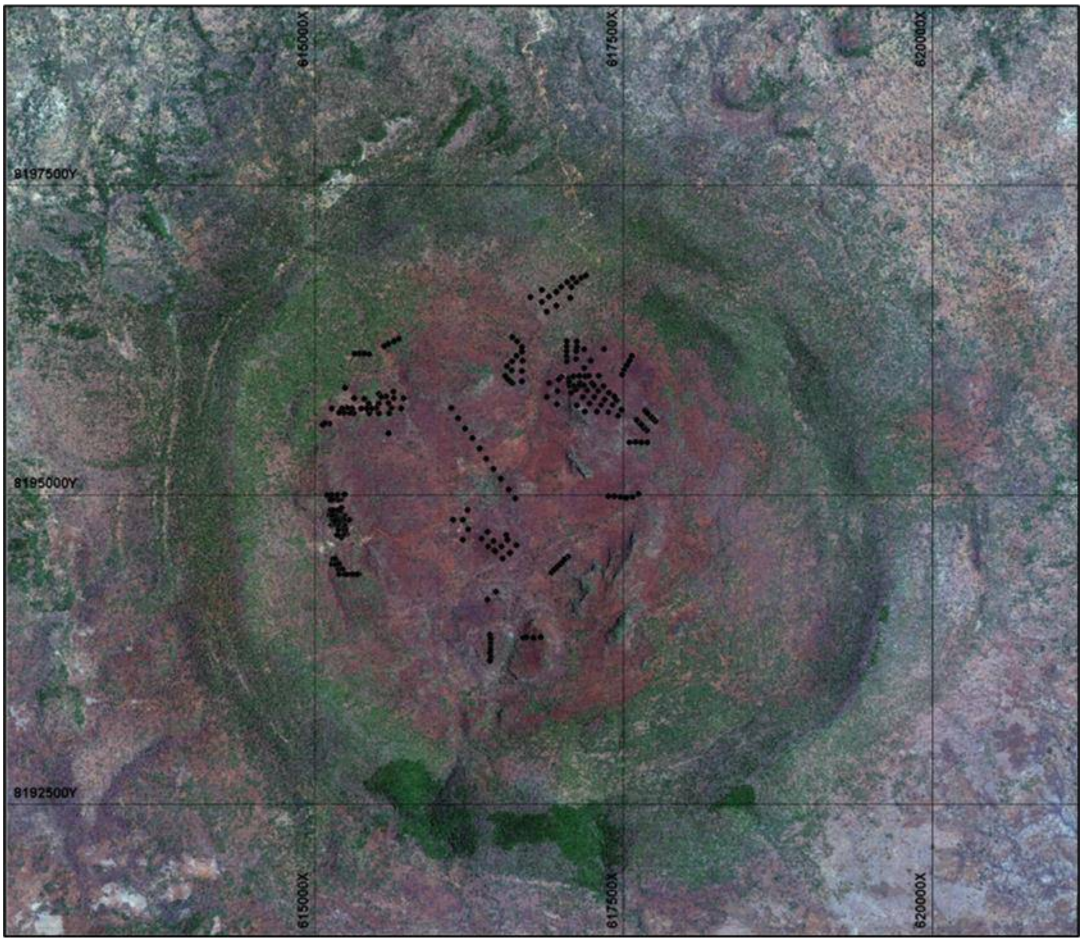
Figure 4.1 Regional geology map



Source: Tambara Sheet no 1634 (GTK Consortium, 2006)

Airborne gamma spectrometry data was useful to support geological mapping, with fenites outcrops corresponding to K anomalies, carbonatites to U and Th anomalies, and sandstones having a very low radiometric response.

Figure 4.2 Satellite image of Monte Muambe, showing the sandstone ridge surrounding the carbonatite/ fenite complex



Note: Drill collars (black dots) are as at time of reporting. Map grid 2.5 x 2.5 km

Source: Altona, 2023

4.2.2 Lithology

Carbonatites

The Monte Muambe carbonatites are fine to medium grained and range from white (fresh) to orange (oxidised) and brown in colour (Figure 4.3), sometimes green. Outcrop surfaces tend to be grey or yellow-brown. Various mineral segregations, including apatite and Fe minerals forming pseudomorphs after magnetite, are locally visible. Karstic weathering features are common.

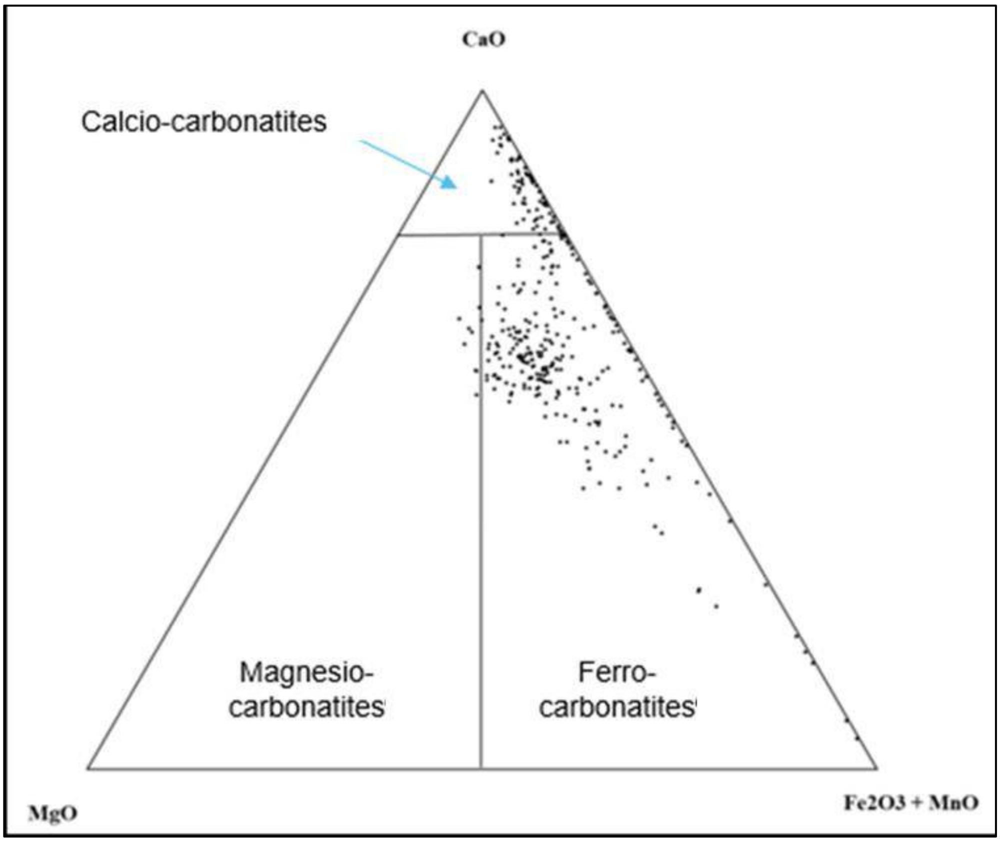
Figure 4.3 Yellow fine grained carbonatite vein intersecting a grey coarse-grained carbonatite

Source: Altona, 2023

The classification of carbonatites is not well established, with one of its latest reviews being that of Yaxley et al (2022). Some descriptive classifications involve geochemical characteristics (Le Bas, 1991) and others the modal percentage of primary (magmatic) carbonate minerals, with varying thresholds (Mitchell, 2005). The grain size has also been traditionally used to distinguish, for example, sövites from alvikites (Kresten, 1983). Most recent publications on carbonatite classification agree that descriptive classifications are strongly limited compared to emerging genetic classifications.

Using the geochemical classification of Le Bas (1991), the Monte Muambe carbonatites are mostly calcio-carbonatites and ferro-carbonatites (Figure 4.4). The applicability of this classification at Monte Muambe has its limitations though, as beside carbonates, carbonatites contain a wide variety of accessory minerals including apatite (average 8 wt%, up to 32 wt%), baryte (average 9 wt%, up to 29 wt%), fluorite (average 11wt%, up to 28wt%) and REE minerals. As a result, the sum of the oxides used for the geochemical classification (Fe_2O_3, MnO, MgO and CaO) usually ranges from 50 to 60 wt%.

Figure 4.4 Ternary IUGS carbonatites classification diagram. DD samples having <7wt% SiO_2



Source: Altona, 2023

While few Monte Muambe samples fall within the magnesio-carbonatite field of the IUGS ternary diagram of Figure 4.4, the same diagram shows that there are two distinct carbonatite suites, one having a very low MgO content (average 0.3wt% MgO), and the other one having a MgO content of up to 12.5 wt% (av. 6.1 wt% MgO).

The cut-off between both suites is at about 1wt% MgO. There are no visible differences between the two suites. For logging purposes, the suites were named Ca-carbonatite and Mg-carbonatite respectively. Preliminary assays using the Company's portable X ray fluorescence machine (pXRF) on site allowed the logging geologist to identify both suites and to log them accordingly. Ca-carbonatites were logged as CCA and Mg-carbonatites as CMG.

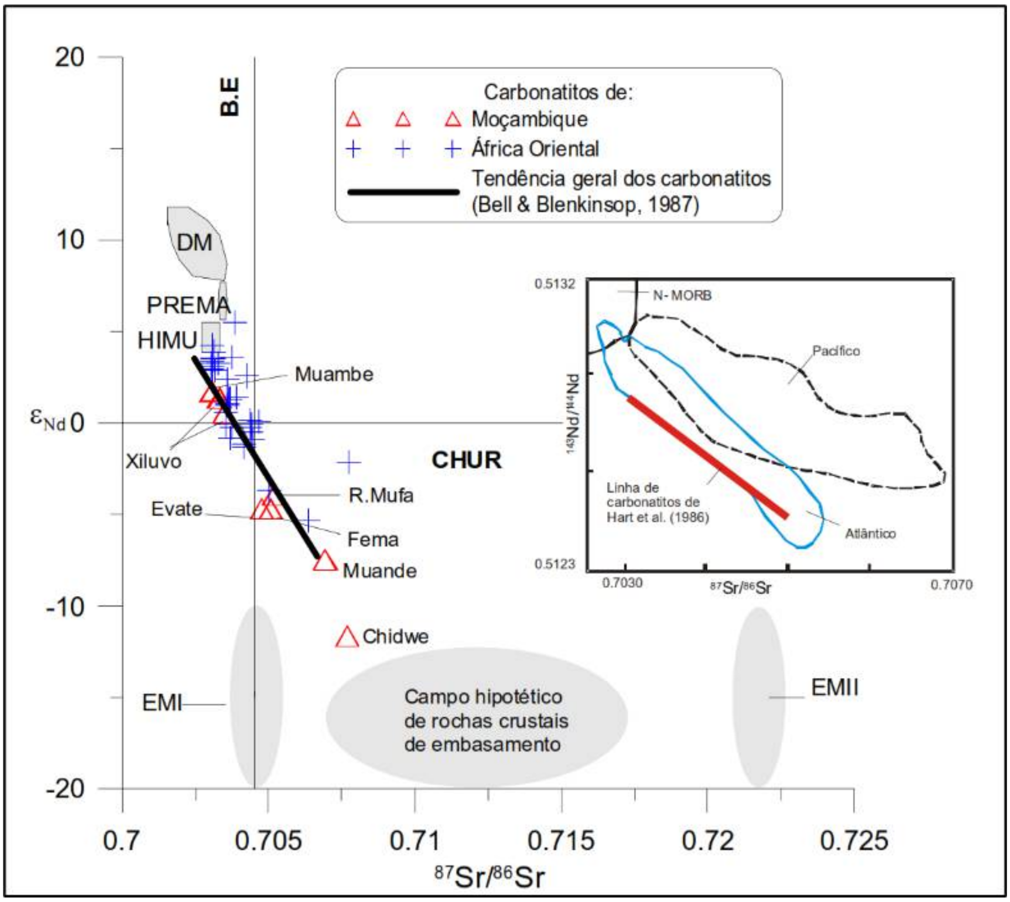
X-ray diffraction (XRD) analysis results show that the main carbonate in Ca-carbonatites is calcite, whereas in Mg-carbonatites it is ankerite.

Chauque (2008) reports stable isotope data for one sample from Monte Muambe (Table 4.1). This data is typical of continental shield Phanerozoic carbonatites and consistent with data from other Mozambican carbonatites (Figure 4.5)

Table 4.1 Stable isotope data for Monte Muambe

ID	$^{87}Sr/^{86}Sr$	$^{143}Nd/^{144}Nd$	$\varepsilon_{Nd(0)}$
Sample 010/06	0.703062	0.512256	1.77

Figure 4.5 Stable isotope data for select carbonatites



Note: Select carbonates from Mozambique and Eastern Africa regions, including Monte Muambe are presented above

Source: Chauque, 2008

Fenites

Fenites are rocks formed through metasomatic interactions between carbonatites and their host rocks (Elliott et al, 2018), and are characterized by a relative enrichment in alkali and silica compared to their protolith. At Monte Muambe, fenites form a 600 to 1,200-m wide aureole along the contact between the carbonatite intrusion and its host Karoo sandstones. Blocks of fenites also occur as xenoliths of various sizes in the carbonatite.

Fresh fenite is buff in colour and coarse grained. It consists mainly of K-feldspar, goethite, and phlogopite. XRF data show that accessory minerals include gorceixite (a Ba-Al phosphate), fluorite, Mn minerals, pyrochlore, apatite and REE minerals. Weathered fenite is typically orange in colour and friable. K-feldspar is replaced by clay minerals, and fine-grained phlogopite aggregates are more visible than in fresh fenite.

Fenites were logged as FEN. For logging purposes, typical fenites are considered to have SiO_2>20wt%.

Carbonatite-fenite relations, and mixed lithologies

Due to the intimate association of both types of rocks, geochemical assays of drilling samples often correspond to a mix of carbonatites and fenites.

This happens in the following situations:

- Contact between carbonatite and fenite (RC samples).
- Stockwork of cm or mm size carbonatite veins in fenite (both DD and RC samples).
- Fenite xenoliths in carbonatite (both DD and RC samples).

Many entries in the RC assay logs correspond to an actual mixture of carbonatite and fenite. The distinction between the two suites of the carbonatite component can still be made based on the MgO content. Mixed lithologies were therefore logged as MCA and MMG respectively.

Sandstones

Sandstones forming the Monte Muambe ridge belong to the Cádzi formation, a full description of which is available in GTK Consortium (2006). The sandstones are immature, arkosic, and sometimes present cross-stratifications. The grains are moderately sorted and subangular to rounded.

Outcrops are relatively rare except in stream beds, but the slopes of the Monte Muambe ridge are covered in sandstone blocks. On the northern side of Monte Muambe, the dip the sandstones appear to be sub-horizontal, as evidenced by horizontal paleo-surfaces and outcrops. On the southern side of the mountain, based on the regional geological map, the dip is expected to be about 10° to 15° in a southerly direction. Sandstones were not encountered in drill holes.

Agglomerates

The term agglomerate has been used in previous maps, based on observations of outcrops. True agglomerates (rocks consisting of large, coarse rock fragments associated with lava flows ejected during volcanic eruptions) have not been observed at Monte Muambe and it is thought that the term has actually been used for various types of breccia and for pyroclastic deposits (see below).

Breccia

Various types of breccia are encountered at Monte Muambe. During logging DD cores a distinction was made between breccias having a fenite matrix (FBR) and breccia having a carbonatite matrix (CBR). Hydraulic breccia with fenite and/or carbonatite clasts can be observed in various parts of the intrusion.

Pyroclastics

Pyroclastic deposits outcrop in the southern part of the Monte Muambe basin. They were also encountered in DD hole MM007 in the northern part of the basin. Particles vary in size from a few mm to a few cm, with occasional dm size igneous and crustal xenoliths, and are often heterogeneous. The matrix is very fine grained. These pyroclastic deposits therefore consist of a mix of ashes and lapilli. They were probably preserved in downward faulted blocks at the base of the now eroded volcanic edifice. Pyroclastics were logged as CPY and OPY depending on whether a carbonatite component is present or not.

Mafic dykes

Mafic dykes were occasionally intersected by drill holes and are also observed on outcrops. Core samples are weathered to very weathered, and it is therefore difficult to identify these rocks on the basis of their geochemistry. The rock is very fine grained, with white phenocrysts in some samples. The SiO_2 content typically ranges from 40% to 50%, and Na_2O + K_2O from 7 to 12% (with some samples being as low as 2%). The Nb/Y ratio is high (3 to 10). This places these rocks in the field of basanites and nephelinites.

4.3 Mineralisation

4.3.1 Carbonatite-hosted REE mineralisation model

REE deposits typically occur in the following geological contexts (Wang, 2013):

- Carbonatites, and residual deposits resulting from the in-situ weathering of carbonatites.
- Alkaline intrusions.
- Ionic clay deposits forming from the weathering of a REE-bearing source rock (typically REE enriched igneous rocks, but sometimes also sediments).
- Placer heavy mineral sands deposits containing REE minerals such as monazite or xenotime.

- Iron oxide copper gold (IOCG) deposits.

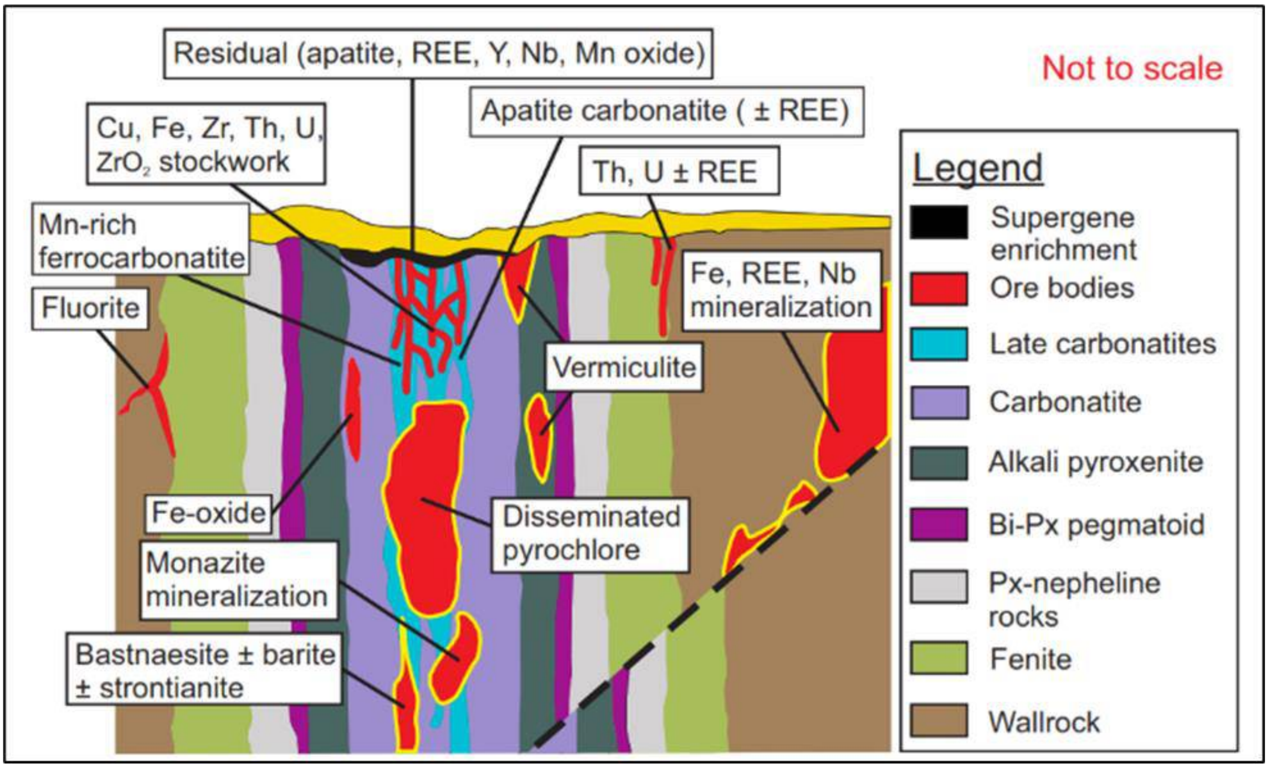
The carbonatite REE deposit model is described by Verplanck et al (2014) and Simandl (2015). It is important to note that while most of the World's REE production originates from a small number of carbonatite-hosted REE deposits, not all carbonatites host potentially exploitable REE mineralisation. Additionally, REE mineralisation is typically the result of various metallogenic processes starting from the magmatic stage, continuing in late and post magmatic hydrothermal stage, and ending with supergene alteration processes (Harmer and Nex, 2016, Anenburg 2020). As a result, the characteristics of carbonatite-hosted REE deposits vary significantly from one to the other, and it is fair to say no two deposits are identical. Even within the same province (for example the Chilwa REE Province – see next section of this report) variations in size, geometry, petrology, REE and other accessories mineralogy, weathering and metallurgy are observed (see a review of Southern and Eastern Africa carbonatite-hosted REE deposits in Harmer and Nex, 2016). This ultimately results in significant differences in mining and processing methods for REE mines.

In practice, there is a notable difference between:

- Primary carbonatite deposits, originating from magmatic and hydrothermal processes, as well as in-situ weathering – these deposits can extend vertically to depths of several hundred metres below the surface (ex Kangankunde and Songwe in Malawi, as well as Monte Muambe).
- Residual deposits resulting from the deep weathering of an underlying primary carbonatite, and forming a blanket over it, typically 10 to 50 m in thickness (e.g., Mrima in Kenya, Nguala in Tanzania, Longojo in Angola and Mt Weld in Australia).

A conceptual model of a carbonatite-hosted REE deposit is presented in Figure 4.6.

Figure 4.6 Vertical section of a hypothetical carbonatite mineralizing system



Source: Simandl, 2015

The most significant REE minerals in carbonatite-hosted REE deposits are bastnaesite, monazite, synchisite, florencite and apatite (Table 4.2). Carbonatites can host other minerals and metals of economic interest, including niobium, copper, barite, fluorite, vermiculite, phosphates, iron and manganese.

Table 4.2 Main REE minerals in African carbonatite deposits

Mineral name	Formula	Contained REO% (typical)
Bastnaesite	$RECO_3F$	76
Florencite	$REAl_3(PO_4)_2(OH)_6$	32
Monazite	$(RE,Th)PO_4$	71
Synchisite	$CaRE(CO_3)_2F$	51

4.3.2 Regional Chilwa REE Province

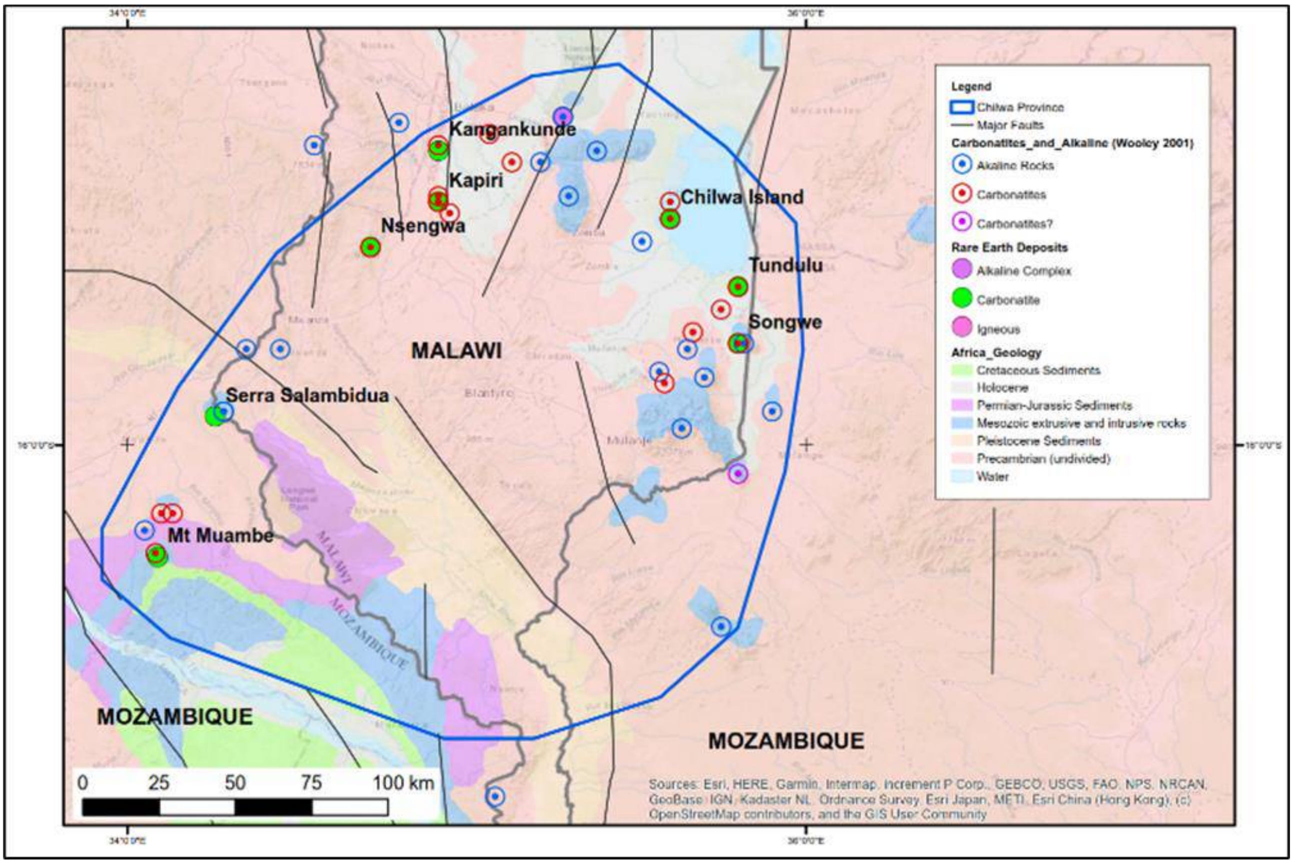
Regionally, intra-continental carbonatite provinces are associated with large igneous provinces and rift systems (Ernst, 2010, Harmer and Nex, 2016). Most carbonatites in Africa are Cretaceous or more recent, but several periods of carbonatite formation are noted in the Neoproterozoic, Mesoproterozoic and Paleoproterozoic (Chauque, 2008).

The Monte Muambe carbonatite has not been dated yet. Geological evidence shows that it is post Upper Karoo. It is usually considered as being a part of the Chilwa Alkaline Igneous Province (Woolley, 1991) (Figure 4.7) which encompasses a considerable number of occurrences of alkaline rocks and carbonatites, while mafic rocks are extremely scarce. It is in this province that carbonatites were recognized and described in Africa for the first time (Woolley, 1991).

The age of the Chilwa Alkaline Province intrusions ranges from 111 to 137 My (Eby et al, 2004), which corresponds to the lower part of the Cretaceous.

The province is characterized by a relative abundance of REE deposits, including the Monte Muambe carbonatite in Mozambique, the Kangankunde and Songwe carbonatites in Malawi, as well as the Chambe clay-associated deposit, also in Malawi.

Figure 4.7 Map of the Chilwa REE Province

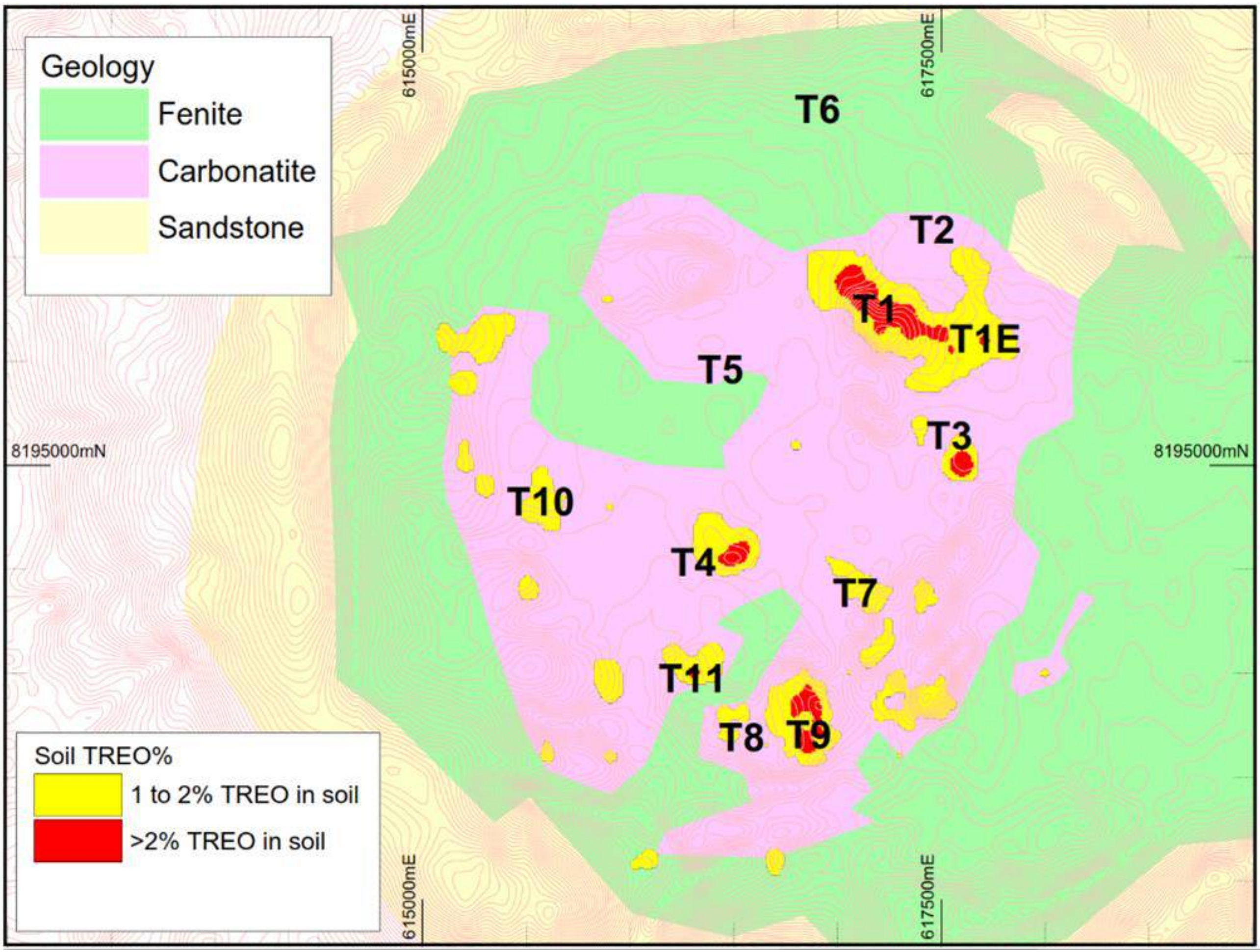


4.3.3 Monte Muambe REE mineralisation

Not all carbonatites carry REE mineralisation at Monte Muambe. Both low grade (0.5 to 1% TREO) and high grade (>1% TREO) mineralisation, as defined further below in this section, are encountered in specific REE-enriched parts of the carbonatite intrusion. Outcropping REE mineralisation is relatively easily identified using the soil sampling survey results, as the soil geochemistry largely reflects the bedrock geochemistry due to the limited thickness of the soil cover (typically 50 cm to 2 m). This does not apply to blind mineralisation such as that of Target 6.

The map of Figure 4.8, therefore, gives a good idea of the repartition of REE mineralisation at the scale of the Monte Muambe carbonatite intrusion. The position of REE mineralisation reflects both primary (magmatic) and secondary (hydrothermal and supergene) processes. Areas with 1 to -2% TREO in soil (yellow), and with >2% TREO in soil (red), roughly correspond to areas with low grade and high grade mineralisation, respectively, in the bedrock.

Figure 4.8 Monte Muambe soil sampling survey results map



Lithology and geochemistry

REE mineralisation at Monte Muambe has been encountered in both Ca-carbonatites and Mg-carbonatites, as well as in fenites. In the fenites, REE mineralisation is likely to be largely carried by small carbonatite veinlets or stockworks, or to originate from hydrothermal remobilization of REE from carbonatites. The later process has been documented at the Songwe carbonatite in Malawi (Broom-Fendley et al, 2021), but more petrological studies will be necessary to ascertain this.

Usually, carbonatite-hosted REE mineralisation in carbonatite complexes tend to occur preferentially in magnesio-carbonatites and ferro-carbonatites (see for example Harmer and Nex, 2016); however, this does not appear to be the case at Monte Muambe based on current data (Figure 4.9).

Additionally, Ca-carbonatites and Mg-carbonatites show substantially the same range of TREO% (Figure 4.9, left), including a marked break around 2.5% TREO between two grade populations (Figure 4.10).

However, two distinct geochemical domains were identified based on their REE and Nb grades (all laboratory assay results) based on their NdPrOx/TREO ratio and of their Nb content (Figure 4.11):

- High grade mineralisation (HGM), with over 1% TREO (av. 2.381%), and a low level of Nb (typically below 500 ppm Nb).
- Low grade mineralisation (LGM), with 0.5% to 1% TREO (av. 0.729%), and a high level of Nb (typically above 500 ppm Nb).

These two domains show a relatively good consistency at the scale of the intrusion and are often juxtaposed. On a ternary map from the airborne geophysical survey data, areas underlain by HGM tend to be associated to thorium anomalies, and areas underlain by LGM tend to be associated to uranium anomalies. This is particularly true at Target 1 and Target 4.

The geochemical characteristics of both domains are summarized in Table 4.3.

Figure 4.9 TREO% vs MgO% (left) and TREO% vs Fe2O3% (right) – all laboratory assay results

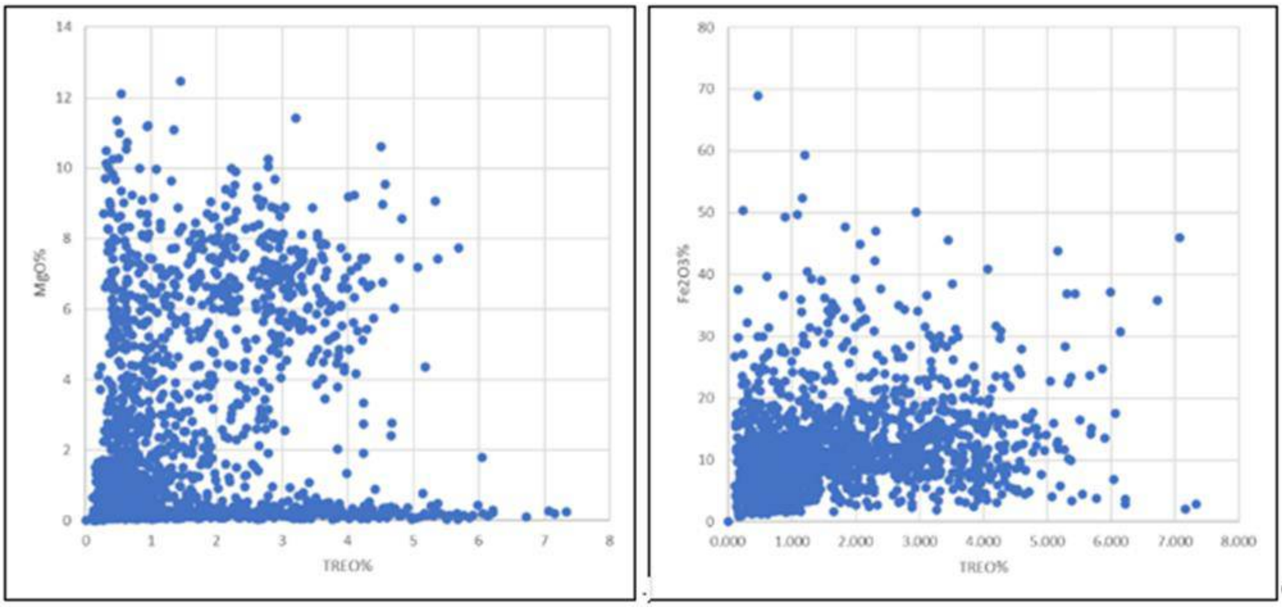
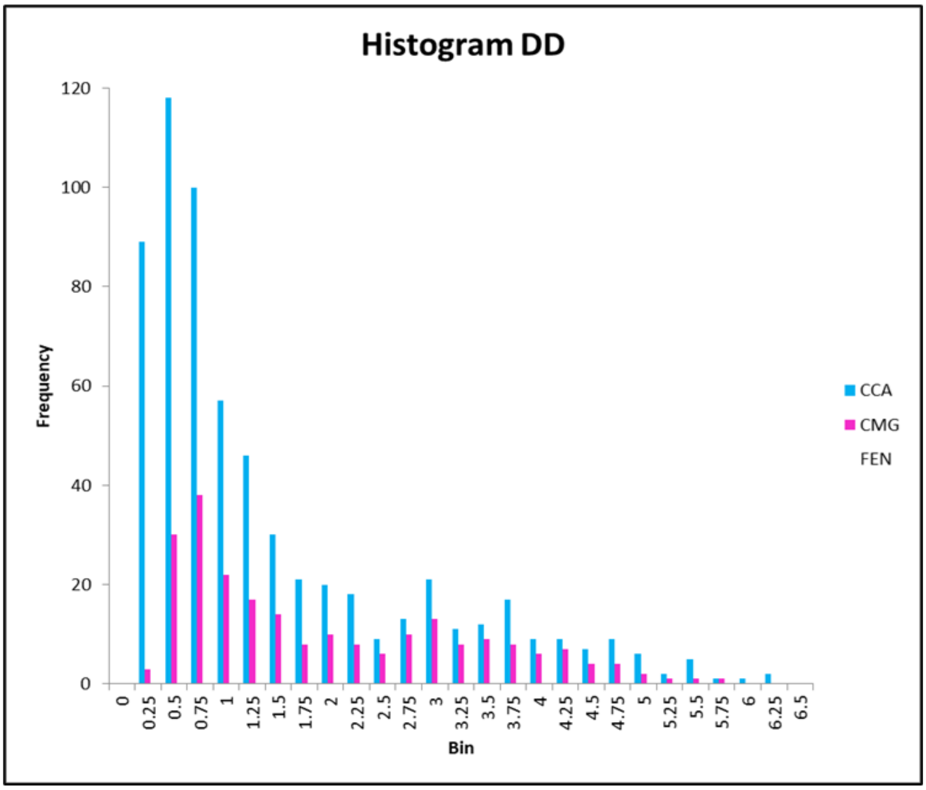


Table 4.3 Average REE, Nb, Th and U content of HGM and LGM domains (all laboratory assay results)

		TREO%	Ce_ppm	Dy_ppm	Er_ppm	Eu_ppm	Gd_ppm	Ho_ppm	La_ppm	Lu_ppm	Nd_ppm	Pr_ppm	Sm_ppm	Tb_ppm	Tm_ppm	Y_ppm	Yb_ppm	Nb_ppm	Th_ppm	U_ppm
HG >1% TREO	Min	1.000	3122	16	8	16	39	3	807	1	704	272	69	4	1	89	4	6	18	2
	Max	7.333	29323	460	284	179	479	98	20439	36	7578	2719	721	66	42	3095	244	5266	1662	168
	Mean	**2.381**	**9215**	**91**	**44**	**55**	**134**	**16**	**6570**	**5**	**2109**	**782**	**221**	**17**	**6**	**499**	**36**	**497**	**290**	**20**
	SD	1.197	4926	47	24	23	60	9	3942	3	1033	398	95	8	3	285	19	597	206	15
LG 0.5-1% TREO	Min	0.500	1369	13	7	11	21	3	399	1	384	142	43	2	1	91	5	4	14	1
	Max	0.999	4288	258	123	98	258	48	3056	1448	2399	609	341	46	15	1267	102	6902	722	150
	Mean	**0.729**	**2614**	**72**	**33**	**38**	**98**	**13**	**1495**	**6**	**879**	**268**	**133**	**13**	**4**	**359**	**25**	**780**	**164**	**16**
	SD	0.148	587	32	15	14	40	6	459	59	240	63	46	6	2	173	12	656	117	13

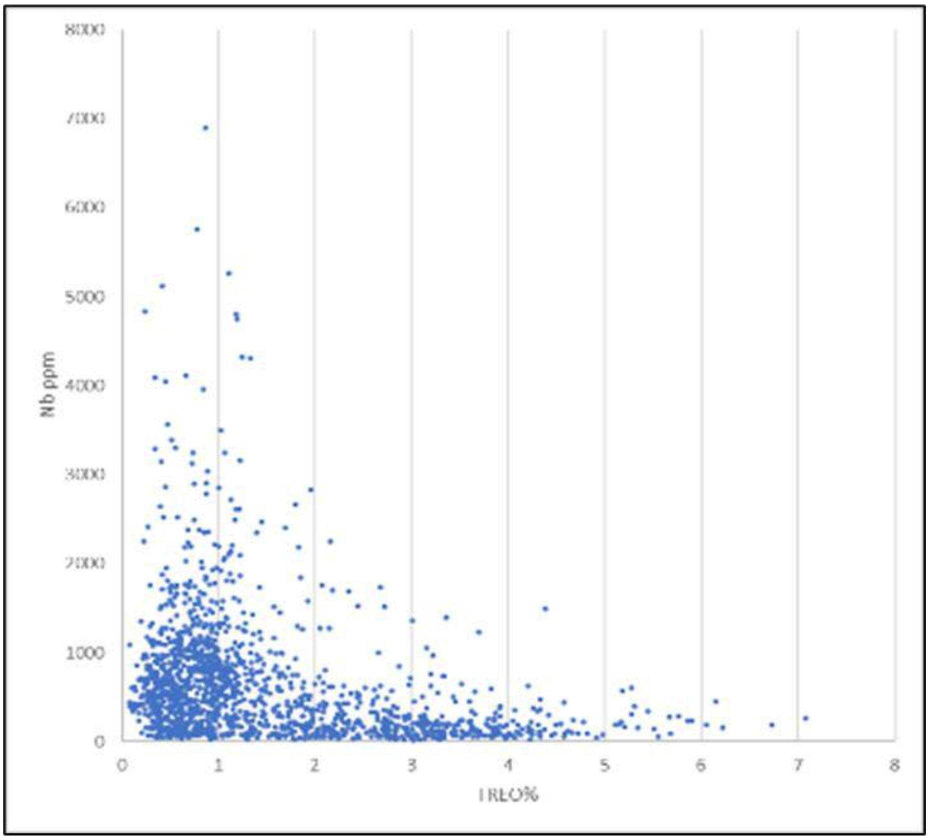
Note: SD – Standard deviation; TREO – Total Rare Earth Oxide

Figure 4.10 TREO% histogram for CCA and CMG carbonatites



Note: These are DD assay results. There is a break between two grade populations at 2.5% TREO.
Source: Snowden Optiro, 2023

Figure 4.11 TREO% vs Nb ppm, 2 m RC composites



Source: Snowden Optiro, 2023

REE mineralogy

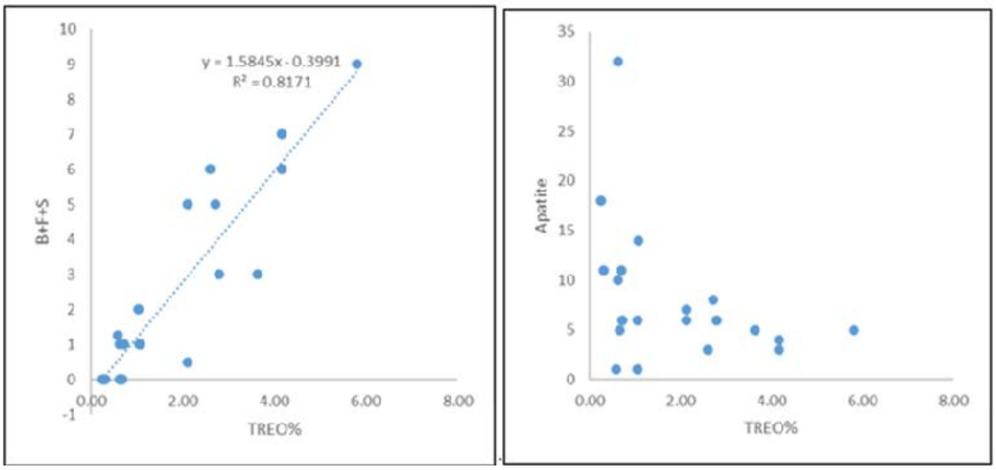
In 2022, the Company submitted 20 samples to Intertek Laboratory in Perth for XRD analysis and geochemical analysis. The samples were selected to cover a range of TREO content, lithology, weathering, and chemical composition. Results are summarized in Table 4.4.

These results show that REE are contained in three minerals at Monte Muambe, namely bastnaesite (the most common), florencite and synchisite. There is a good correlation between the sum of these three minerals expressed as wt% and the TREO% from geochemical assays of the same samples (Figure 4.12), and no correlation between Apatite wt% and TREO%, indicating that no or little REE are contained in apatite. The sample is a carbonatite containing 1.549 wt% TREO. REE mineral form agglomerates ranging in size from 0.5 to 2 mm. This is confirmed by observations on core X-ray fluorescence (XRF) maps (see below).

The two samples containing synchisite as the only REE mineral have a high proportion of Nd and Pr, with a NdPrOx/TREO ratio of 0.44. For comparison, the NdPrOx/TREO ratio for samples containing bastnaesite and/or florencite is 0.17. Of the three samples containing florencite, two belong to the fenite domain. Niobium in LGM is contained in pyrochlore (Table 4.4).

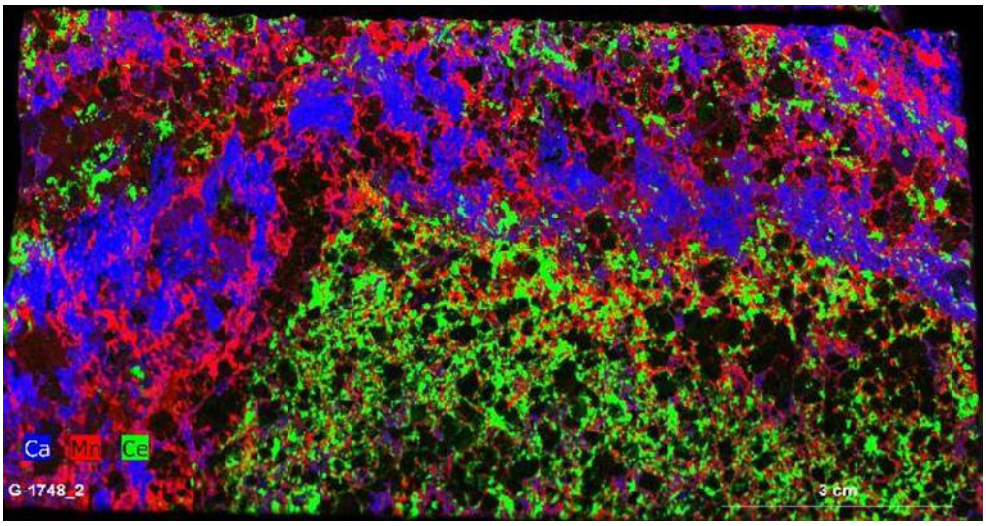
Several core samples were submitted to Dr Hamed Pourkhorsandi of the Université Libre de Belgique in Brussels for mineralogical studies. As part of this work, XRF mapping of half core samples was undertaken. The results are preliminary in nature but give an idea of the distribution of REE minerals in these samples (Figure 4.13 and Figure 4.14). Core sample G1746 (drill hole MM001, 16.62 to 17.54 m) is a carbonatite containing 0.682 wt% TREO. REE mineral form agglomerates ranging in size from 200 µ to 1.3 mm, as well as fissure fillings.

Figure 4.12 TREO% vs bastnaesite + florencite +synchisite wt% (left); TREO% vs apatite wt% (right)



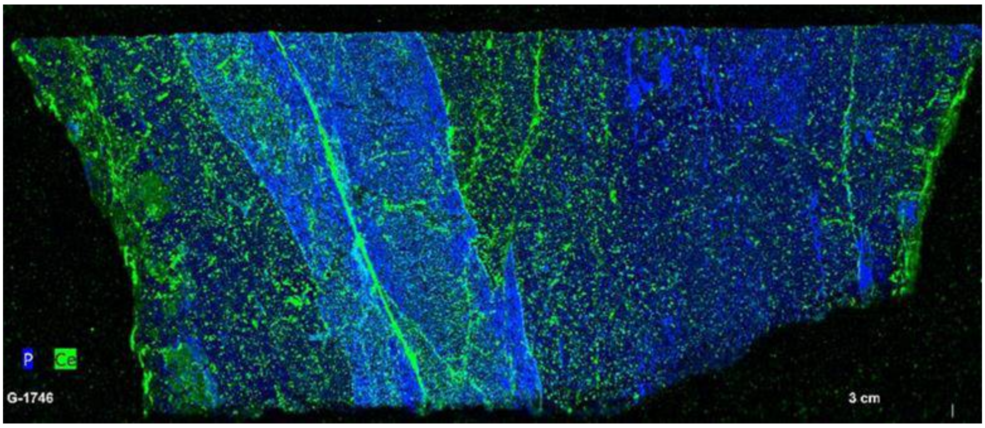
Source: Snowden Optiro, 2023

Figure 4.13 Example of XRF map (Ca, Mn, Ce) of core sample G1748



Note: XRF map for drill hole MM001, 63.43 to 63.64 m.
Source: Altona, 2023

Figure 4.14 Example of XRF map (P Ce) of core sample G1746



Note: XRF map for drill hole MM001, 16.62 to 17.54 m
Source: Altona, 2023

Table 4.4 2022 XRD Analysis results summary (results in wt%)

Sample ID	Domain	LITHO	TREO %	NdPrOx %	Apatite	Bastnaesite	Florencite	Synchysite	Pyrochlore	Amorphous Content*	Ankerite	Calcite	Dolomite	Fluorite	Barite	Celestite	Gorceixite	Goethite	Hematite	Magnetite	Pyrite	Cryptomelane	Pyrolusite	Expanding clay**	Mica*	Opaline Silica	Potassium Feldspar	Quartz	Siderite	Strontianite
B8201	HG	FEN-Weath	2.12	0.37	7	0.25	0.25		<0.5	30	<0.5	2		9				11	2			2	<0.5	6	18		10	1		<0.5
B8202	HG	CCA-Weath	4.16	0.59	3	6				19		38		16				9	1			4	<0.5					1		
B8203	HG	CCA	0.69	0.15	11				<0.5	14		59		5				9							<0.5			<0.5		
B8204	HG	CCA	0.61	0.11	10				<0.5	15		53		15				5							1			<0.5		
B8205	HG	CMG	2.12	0.22	6	5			<0.5	19	6	1		2	1	1		1	14	5			3		<0.5			<0.5	34	
B8206	HG	CMG	0.31	0.05	11				<0.5	17	28	3		14				9	15			1						1		<0.5
B8207	HG	CMG	0.72	0.15	6	1			<0.5	18	29	2		19				13	7									<0.5		
B8208	HG	CMG	1.04	0.16	6	2				16	48	2		14		5		3	1									<0.5		<0.5
B8209	LG	CMG	0.25	0.04	18					17	23	16		2	1	1		12	2			2						2		3
T8156	LG	CCA	1.06	0.21	14	1			1	15		55		7				1	3						1			<0.5		
Z8218	HG	FEN	2.60	0.34	3	6				19		5		21	7			9	2		<0.5	3						25		
Z8281	HG	CCA	5.81	0.63	5	8	1			22		2		23	29			3	1			3						2		
Z8526	HG	CMG	4.16	0.52	4	7				22		10	20	17	3			9	3		1	2	2					<0.5		
Z8563	Waste	FEN	2.80	0.44	6	1	2		2	28	<0.5			4			6	16	2			2	1		10	5	11	1		
Z8569	Waste	FEN	0.58	0.10	1	0.25	1		<0.5	22				8			2	6	2			2	1	<0.5	5	4	45	1		
Z8763	**HG**	**CCA**	**2.72**	**0.33**	**8**	**5**			**<0.5**	**29**		**<0.5**		**28**	**5**			**12**	**6**		**1**	**4**			**<0.5**			**1**		
Z8820	**LG**	**CMG**	**0.65**	**0.14**	**5**				**1**	**16**	**4**	**50**		**2**				**1**	**2**	**1**		**2**			**1**			**1**	**11**	**1**
Z8926	**LG**	**CCA**	**3.64**	**1.57**	**5**			**3**		**31**				**2**	**19**			**18**	**6**			**4**	**9**				**2**	**1**		
Z8955	**LG**	**CCA**	**1.05**	**0.46**	**1**			**1**		**30**		**1**		**3**	**12**			**27**	**11**			**4**	**8**				**1**	**1**		
Z8989	**LG**	**CCA**	**0.62**	**0.15**	**32**	**1**			**1**	**20**		**19**	**2**	**3**	**1**			**7**	**6**	**<0.**		**1**		**<0.5**	**1**		**1**	**<0.5**	**1**	**<0.5**
Average HG	**HG**		**2.25**	**0.30**	**7**	**4**	**1**			**20**	**28**	**16**	**20**	**15**	**9**	**3**		**8**	**5**	**5**	**1**	**3**	**3**	**6**	**10**		**10**	**5**	**34**	
Average LG	**LG**		**1.21**	**0.43**	**13**	**1**		**2**	**1**	**22**	**14**	**28**	**2**	**3**	**8**	**1**		**11**	**5**	**1**		**3**	**9**		**1**		**1**	**1**	**6**	**2**

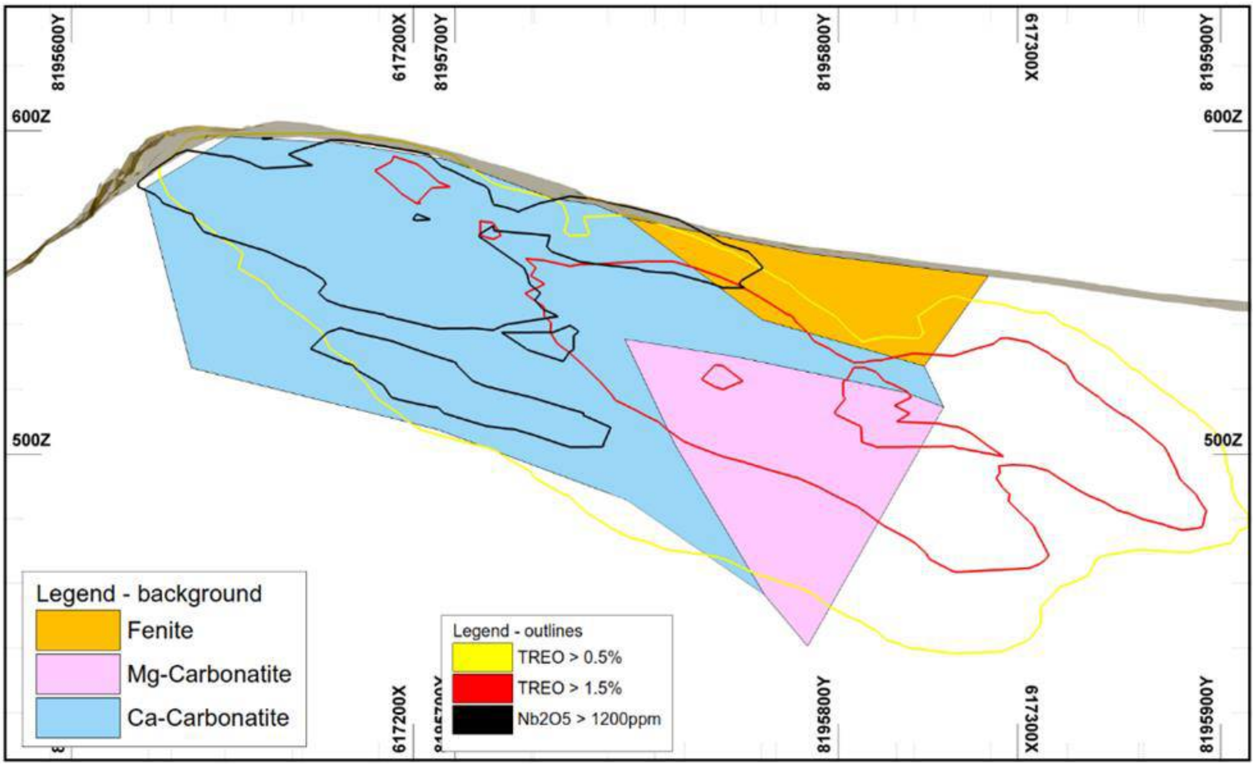
REE mineralisation geometry – Target 1

The core part of Target 1 consists of a 500 m long HGM zone, 40 to 80 m thick, and dipping towards the NE. The dip angle varies from 35° (central part) to 50° (NW and SE parts).

The following observations can be made (Figure 4.15 and Figure 4.16):

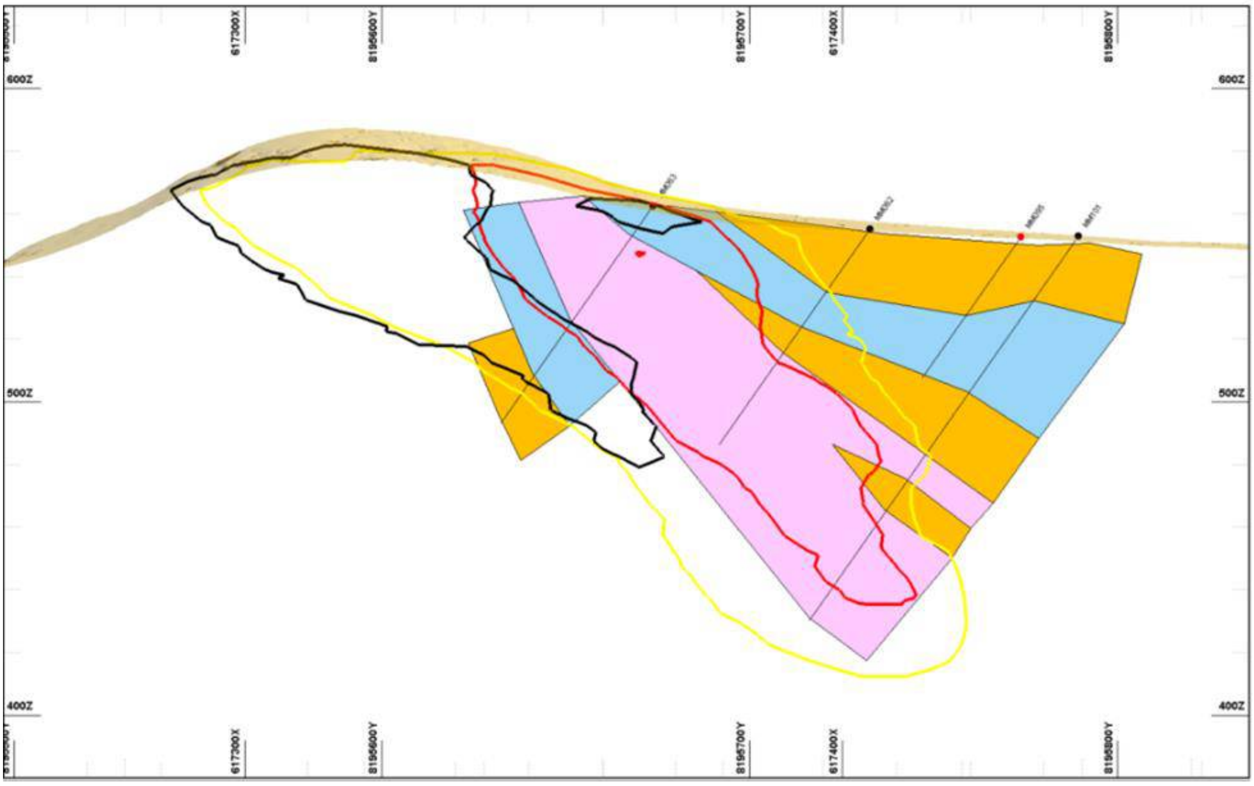
- The hanging wall of the HGM zone, where it does not outcrop, often corresponds to or is close to the foot wall of overlying fenites.
- HGM occurs in both Ca-carbonatite and Mg-carbonatite and across the boundary between these two lithologies.
- LGM (Nb_2O_5> 1,200 ppm and TREO <1%) occurs mostly in Ca-carbonatite and forms a 20 to 60 m thick zone at the footwall of the HGM zone. LGM is also more rarely present at the hanging wall of the HGM, especially in the SE part of Target 1.
- In Figure 4.15 the cross-section passes through drill holes MM058 and MM097. Grade outlines correspond to grade shells from the block model (red polygons = 1.5% TREO cutoff; black polygon = 1,200 ppm Nb_2O_5 cutoff). In Figure 4.16 the cross-section passing through drill holes MM063 and MM101.

Figure 4.15 SW-NE cross section of the central part of Target 1



Source: Snowden Optiro, 2023

Figure 4.16 SW-NE cross section of the southern part of Target 1

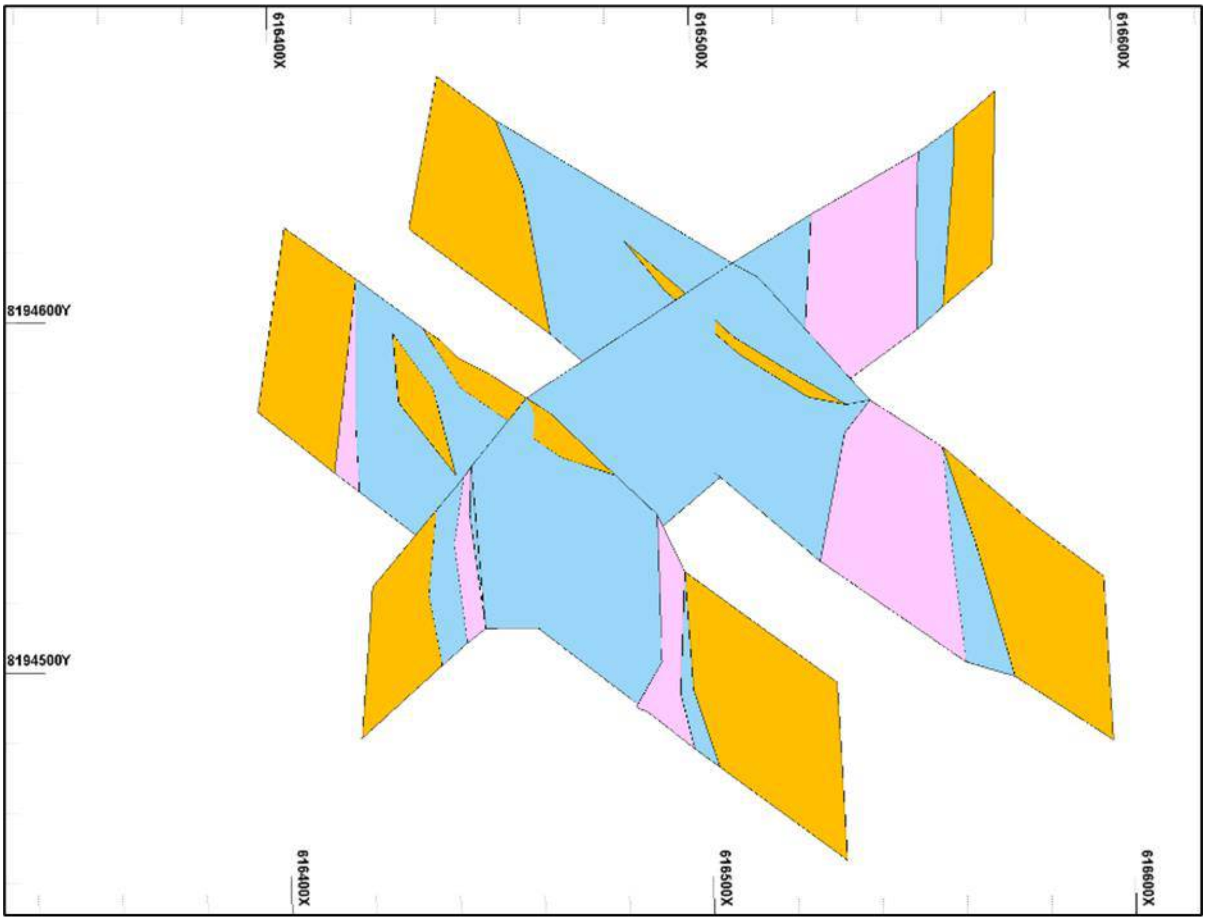


Source: Snowden Optiro, 2023

REE mineralisation geometry – Target 4

REE mineralisation at Target 4 is associated to a sub-vertical carbonatite pipe hosted in fenite and having a diameter of 130 to 170 m at surface level. The pipe consists mostly of Ca-carbonatite, with a ring-shaped Mg-carbonatite dike close to or at the contact between the pipe and the host fenite and pinching out on the NW side (Figure 4.17). As can be seen in Figure 4.18, high grade intercepts are not bound by lithological contacts. HGM forms a zone which is largely associated to the carbonatite pipe, but which does not follow exactly its boundaries.

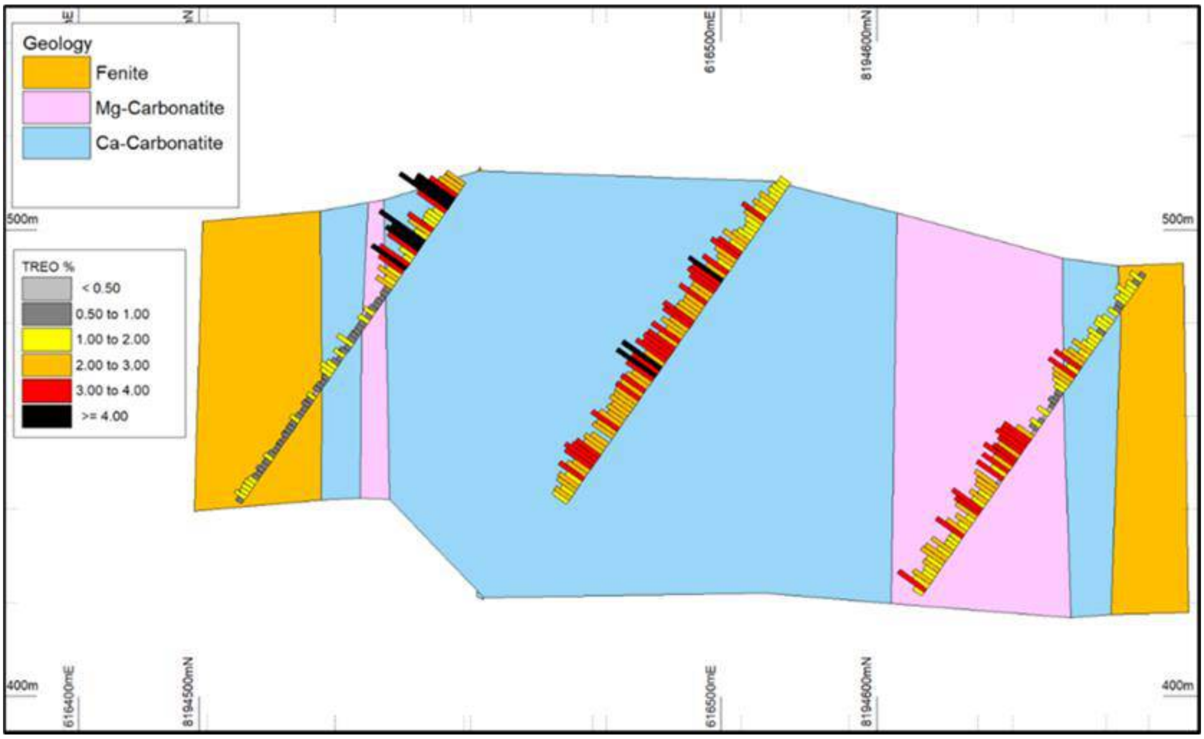
Figure 4.17 Block diagram of Target 4, looking north



Note: Orange: fenite, blue: Ca-carbonatite and mixed Ca-carbonatite/fenite, pink: Mg-carbonatite and mixed Mg-carbonatite/fenite

Source: Snowden Optiro, 2023

Figure 4.18 SW-NE section of Target 4 with borehole logs (TREO%)



Source: Snowden Optiro, 2023

REE mineralisation controls

As observed at Targets 1, 4 and 6:

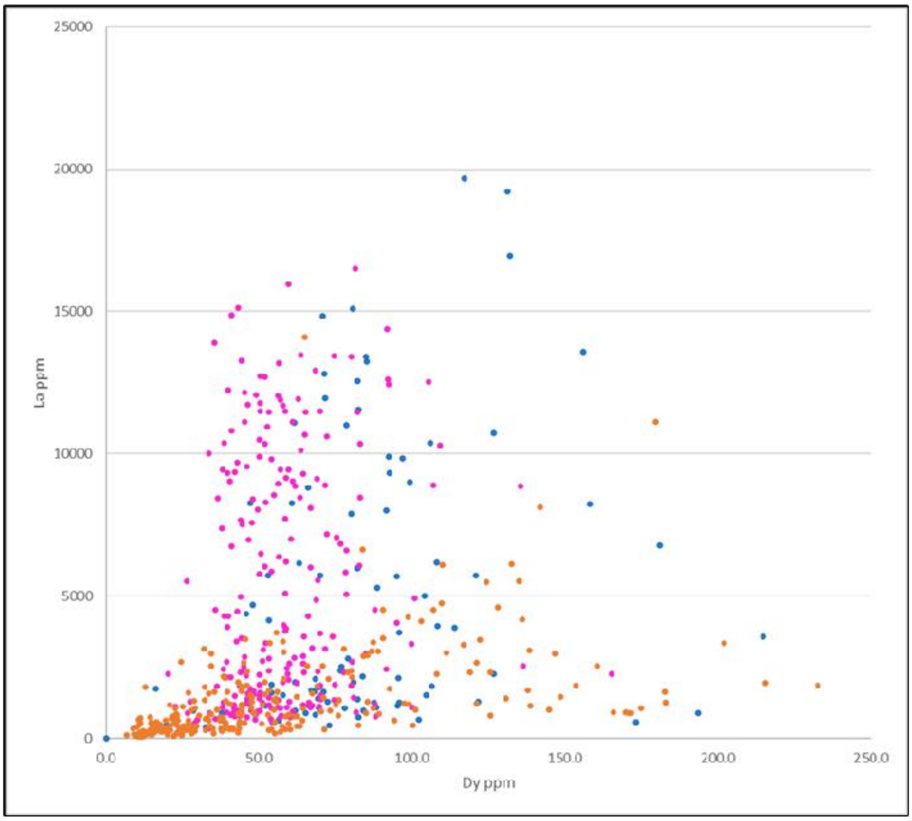
- LGM is mostly associated with Ca-carbonatites.
- HGM is associated with both Ca-carbonatites and Mg-carbonatites, and often extends across boundaries between these two lithologies. This suggests that lithology is not the sole control for HGM, and that the current geometric distribution of REE at Monte Muambe may be the result of post-magmatic remobilisation and redeposition across geological boundaries.
- At Target 1 and at Target 6, HGM seems to be preferentially developed immediately below the contact between carbonatites and overlying fenites (see Figure 4.15 and Figure 4.16), which has a low dip angle, suggesting that the fenites may have acted as a conduit and/or as a cap during remobilisation and redeposition of REE.
- Variations of the NdPrOx /TREO ratio as well as comparisons between light REE and heavy REE; for example La and Dy (Figure 4.19) may indicate that REE fractionation occurred during the process resulting in the currently observable mineralisation. This could involve:
  - Selective remobilisation of light REE (La and Ce in particular)
  - HREE enrichment in fenites (similar to occurrences described in Malawi by Broom-Fendley et al, 2021).
- The 3D inversions of the 1998 helicopter-borne magnetic survey prepared by Altona's consulting geophysicist Joseph Komu in 2021 showed the presence of several Analytical Signal (AS) anomalies (Figure 4.20). Five of these anomalies are relatively shallow, extending from 100 to 700 m below the surface. The other two are much larger and extend down to 2 km below the surface. It is noteworthy that two of the shallow anomalies are located immediately below Target 1 and Target 4 respectively, and that another four targets from the soil sampling survey (Targets 8, 9, 10 and 11) are located immediately above AS anomalies. These anomalies could be related to Fe-carbonatites that may have acted as a source of REE at the scale of the whole intrusion. Testing this hypothesis would require deep drilling below Target 1 and Target 4, although using the location of the anomalies as a prospecting guide and successfully identifying new HGM would also prove it.

Available information therefore points towards HGM resulting from the remobilisation of magmatic REE mineralisation through hydrothermal or supergene processes. Similar processes have been documented and described in other carbonatite associated REE occurrences, for example at the Okurusu fluorspar deposit in Namibia (Cangelosi et al, 2019). This remobilisation would involve fractionation of La and Ce (with a decrease of the NdPrOx/TREO ratio) and would occur in a pervasive manner across different types of carbonatites, with overlying fenites acting as a capping and guiding the geometry of the final mineralisation.

Source carbonatites may correspond to proximal LGM, or to deeper-seated REE mineralisation possibly linked to the AS anomalies mentioned above.

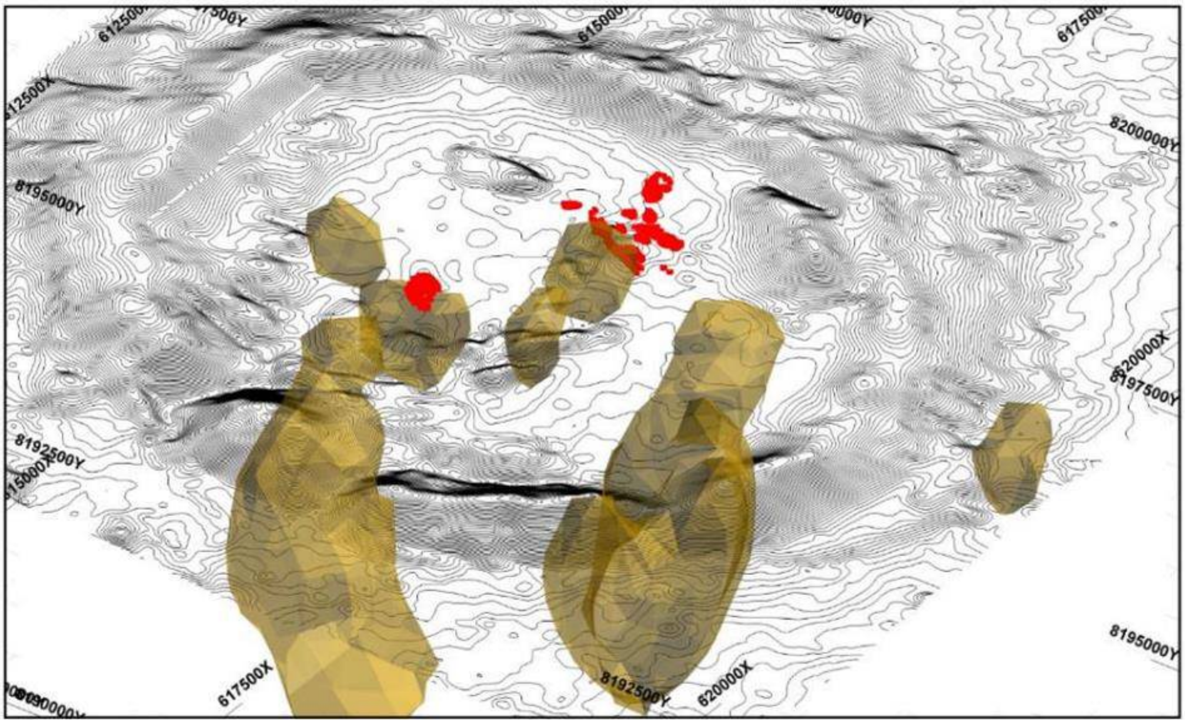
Additional petrological, mineralogical and geochemical work will be necessary to fully understand the REE mineralisation controls at Monte Muambe.

Figure 4.19 Lanthanum vs dysprosium scattergram



Note: DD holes only. Orange: fenite, blue: Ca-carbonatite, purple: Mg-carbonatite
Source: Snowden Optiro, 2023

Figure 4.20 Oblique view of Monte Muambe, looking NNW



Note: Position of AS anomalies is shown in brown. Block models for Target 1, Target 4 and Target 6 (red– TREO>1 %).
Source: Snowden Optiro, 2023

4.3.4 Fluorite mineralisation

Monte Muambe was originally considered as a fluorspar (or fluorite) project, and fluorspar exploration has taken place from the 1960s to the 2010s. The occurrence of fluorspar (CaF_2) in some carbonatite deposits is well documented (Hagni, 2015), and fluorspar is mined from carbonatites in several parts of the world, including the Okorusu mine in Namibia.

Exploration for fluorspar at Monte Muambe culminated with the publication by Globe Metals of an Inferred Mineral Resource (JORC, 2004) of 1.63 Mt of fluorite mineralisation with a grade of 19% fluorite with a cut-off of 10% CaF_2.

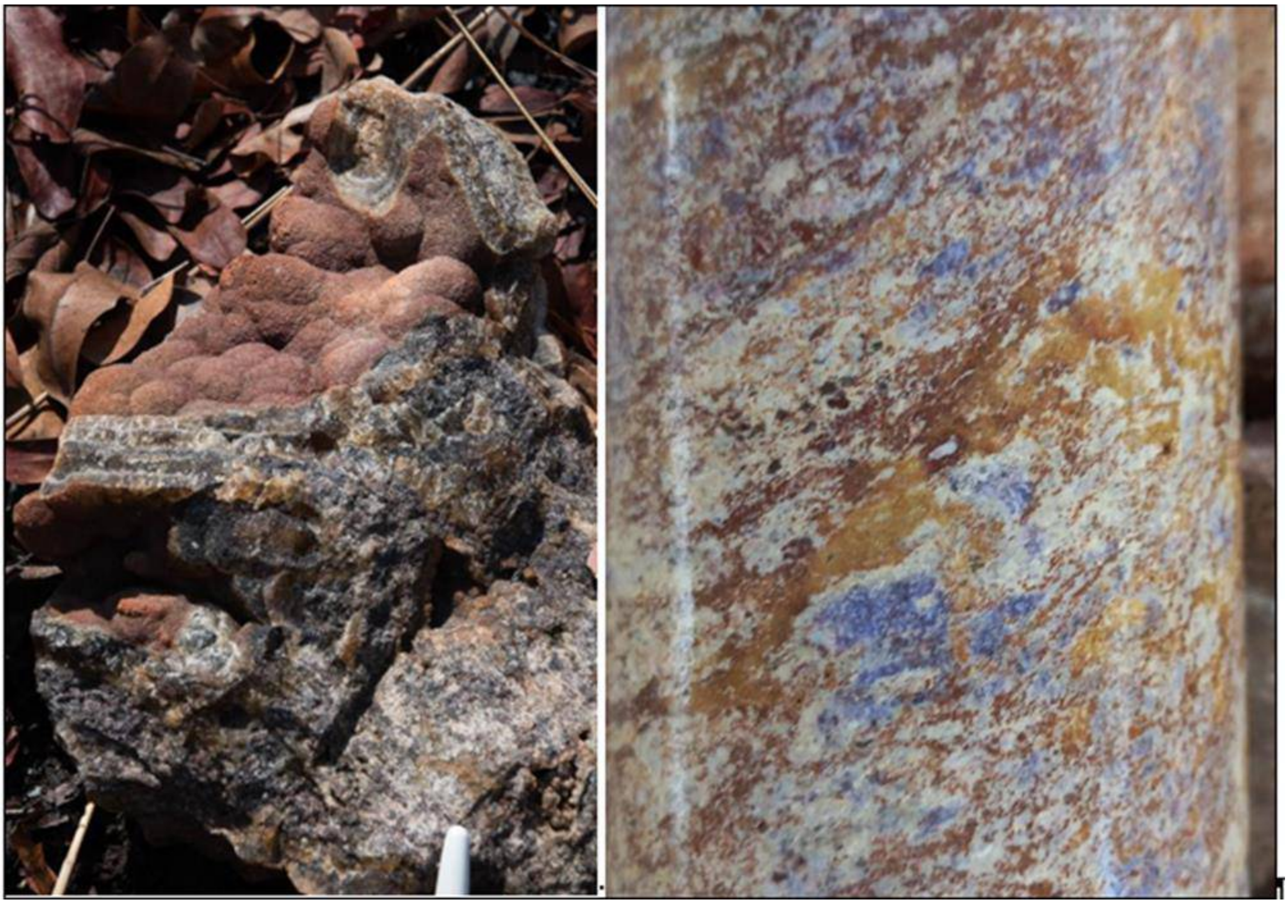
Fluorspar occurs in two types of geological contexts at Monte Muambe (Figure 4.21):

- As an accessory mineral in carbonatites.
- As late hydrothermal veins cutting across carbonatites and fenites.

Fluorspar in hydrothermal veins has a botryoidal habit, which is relatively unusual for this mineral.

No hydrothermal fluorspar mineralisation has been observed during resource drilling at Target 1 and Target 4, although disseminated fluorspar is common. XRD analysis show an average fluorspar content of 15 wt% in the high-grade mineralisation. This is consistent with a 6.52 wt% average fluorspar content in HGM, which corresponds (assuming all the fluorspar contained in fluorite) to 13.4 wt% fluorspar.

Figure 4.21 Botryoidal honey-coloured fluorite from hydrothermal veins (left); anhedral blue fluorite in carbonatite (right)



4.4 Snowden Optiro site visit

The Competent Person's representative, R.N. Barnett, visited the Monte Muambe prospecting site from 7 to 10 August 2013 (two days physically onsite). Key aspects addressed during the site visit were inspection of RC drilling and sampling procedures and density retests. In addition, the survey method, QC sample insertions and assay results, and database control were discussed with the General Manager.

Findings were as follows:

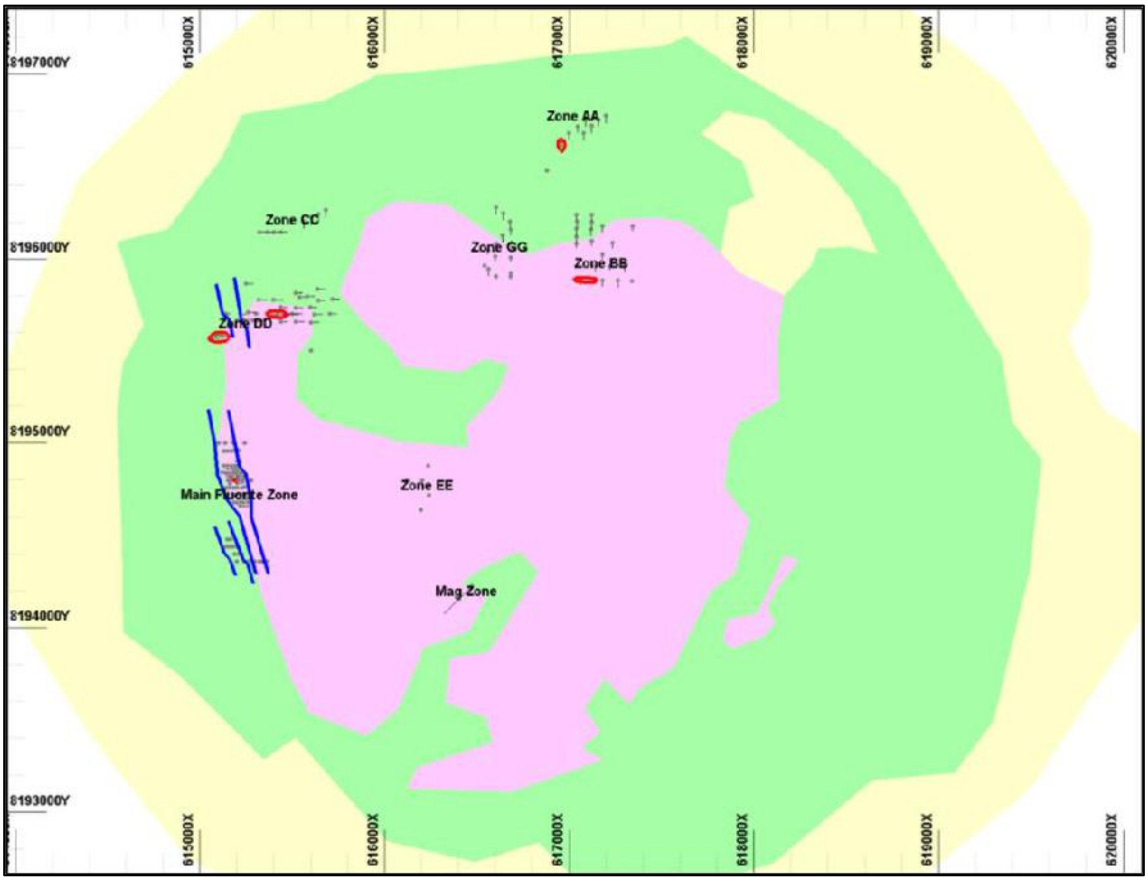
- RC sampling was well controlled and conducted according to the standard operating procedure.
- Density check measurements were conducted on seven core pieces using the standard operating procedure for density adapted for half core and using check weights to check the scale calibration. The retest results were within 2.5% of the original density measurements which is deemed acceptable for resource estimation.
- The survey method used for both borehole co-ordinates and surface digital terrain model uses an internal beacon for control with co-ordinates established using a DGPS system operated by a trained technician.
- Analytical QC insertion rates for both DD and RC samples average at 3.5 to 3.7% which is deemed by the Competent Person to be lower than the generally applied 5% insertion rates for resource estimation. Internal checks comparing 2 m and 3 m composites are deemed, by the Competent Person, to adequately address this issue.
- Core and RC sample storage was inspected with some shortfalls in standards, especially regards RC sample storage. This is not deemed to have impacted on sample security to date, but recommendations were made to improve storage to an acceptable level.
- Database control was inspected. To date the database uses MS Excel spreadsheets controlled by the General Manager with backup on the Company's server. While this has been an acceptable level of control to date it was recommended that improved systems and controls be put in place for future prospecting work.

4.5 Historical and recent exploration

4.5.1 Historical exploration

A detailed discussion of historical exploration is described in section 3.6 of this report. Note that the Globe drilling data from 2009 – 2013 for the fluorite Mineral Resources reported does not lie in the same position as the Altona REE targets (Figure 4.22). These exploration targets were assessed during fluorspar and REE exploration between 2010 and 2012; Globe drilled 165 boreholes in total (including 68 borehole and 5,589 m of drilling not utilised in the fluorite MRE) and the data outside of the fluorite Mineral Resource was utilised for purposes of understanding the deposits, REE target generation and REE target testing by both Globe and Altona. Field mapping is known to have been comprehensively completed by Globe as evidenced from their geological map as seen in Figure 4.1.

Figure 4.22 Exploration targets assessed by Globe Metals and Mining



Note: Grey dots: drill hole collars. Red outlines: collars with REE intercepts. Blue lines: fluorite zones
Source: Altona, 2021

4.5.2 Recent exploration

2021

Altona secured the Monte Muambe as a REE project in June 2021. Initial exploration work was designed to:

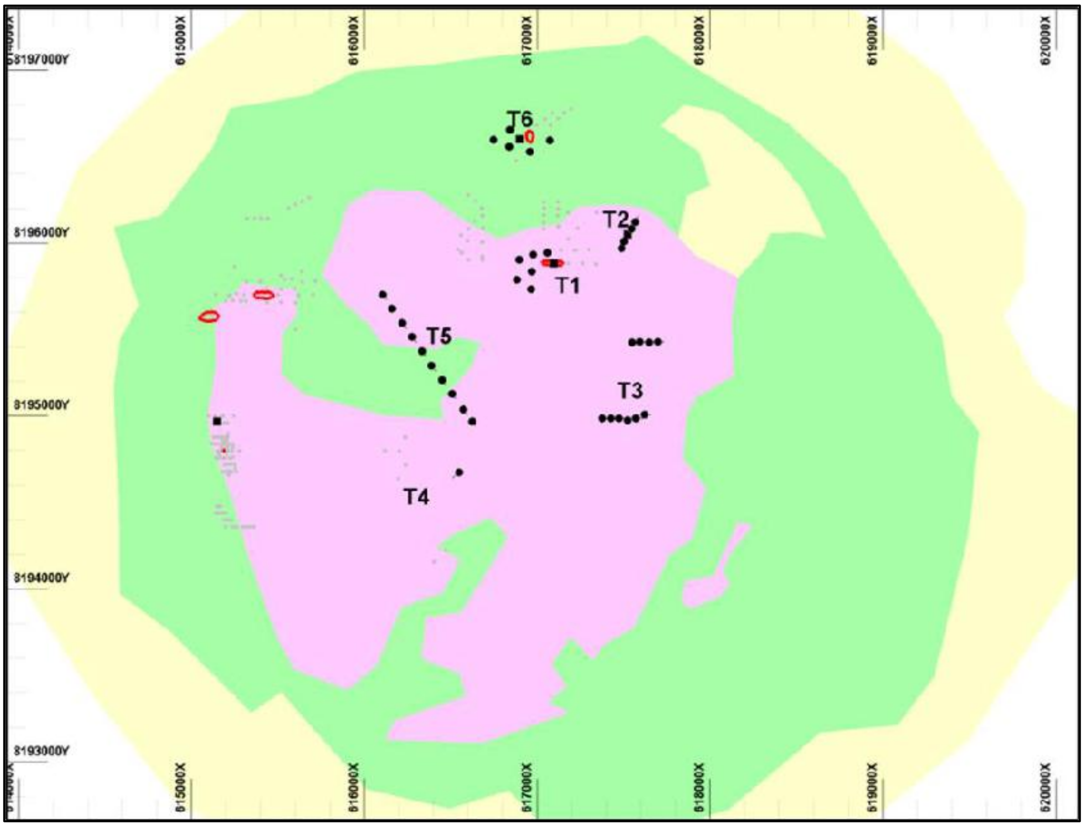
- Test the lateral extension of the most promising REE intercepts from legacy drilling at Target 1 and Target 6 (Figure 4.23).
- Test the possible presence of residual REE deposits in a topographically low part of the Monte Muambe basin (Target 5).
- Test targets based on limited legacy outcrop sampling and soil lines, as well as from legacy (1998) helicopter-borne geophysical survey data (Targets 2, 3 and 4).
- Carry out a DD campaign to help understanding better the geological characteristics and controls of REE mineralisation, and to collect core samples for density measurements.

In total in 2021, five diamond core boreholes were drilled (590.7 m) and 36 RC boreholes (2,518 m). The 2021 drilling campaign showed:

- That Target 1 was not extending towards the west but was potentially extending towards the east.
- That Target 6 was potentially too deep (50 m below surface) to be mineable – it was therefore removed from subsequent exploration plans.
- That no residual REE deposits exist at Monte Muambe – they seem to have been eroded and removed at some point in the history of the deposit.

- The presence of high-grade REE mineralisation (long intercepts with > 2% TREO) at Target 3 and Target 4 (both new discoveries).

Figure 4.23 Altona 2021 REE exploration summary



Note: Black dots are RC collars; black squares are DD holes collars; grey dots are legacy hole collars

Source: Altona, 2021

2022

In February 2022, Altona acquired a portable XRF (pXRF) analyser with the capacity to analyse for La, Ce, Nd, Pr and Y. The use of the pXRF analyser allowed for quasi real time (within 24 hours) geochemical assessment of samples by using the sum of the above five REEs as a proxy for Total Rare Earth Oxides (TREO) per cent. Due to the non-visual occurrence of REEs within rock and soil, the pXRF allowed for real time prospecting decisions and planning to be made.

The first use of the pXRF in 2022 was to carry out a comprehensive soil sampling survey over the Monte Muambe basin, covering carbonatite and fenite outcrops (Figure 4.25). This was carried out initially on a 100 m x 100 m grid, with infill follow up sampling on a 50 m x 50 m and, where required, 25 m x 25 m. A second use for the pXRF is to analyses borehole RC chips and diamond core.

The soil sampling results allowed to define better the outline of Targets 1 and 4, as well as to identify five new targets. The geochemical characteristics of the soil samples allowed for the recognition of the presence of low-grade or high-grade REE mineralisation. The newly identified targets were Target 1E (low-grade), Target 7 (low-grade), Target 8 (low-grade), Target 9 (high-grade), and Target 10 (low-grade).

The use of the pXRF on RC samples allowed the identification of the two relatively consistent types of mineralisation described in section 4.3.3 of this report, i.e. a low-grade mineralisation between 0.5 and 1% TREO with about 0.25% Nb, and a high grade mineralisation with low (less than 500ppm) Nb and an average of 2.5% TREO. The pXRF results also show that there are two carbonatite suites (see section 4.2.2 of this report); i.e., a low Mg (<1% Mg) and a higher Mg (>1% Mg).

In 2022, the Company's exploration drilling strategy was:

- To focus on resource drilling at Target 1 and Target 4, which seemed (and were confirmed to be) the most promising in terms of tonnage and grade. This was initially carried out through drilling 70 m inclined RC holes and, once the dip angle of Target 1 mineralisation was understood, through drilling 120 to 160 m inclined RC holes to explore down-dip. Shallow drilling (25 m deep inclined RC holes) on the low-grade Targets 1E, 7 and 8 was undertaken to test their potential for niobium.

- To test other potential high-grade targets. Reconnaissance work at Target 9 in 2022 (four RC drill holes) was not very successful, with the thickness of the only high-grade intercept being disappointingly low in relation to the size of the soil anomaly. This target still requires additional exploration work.

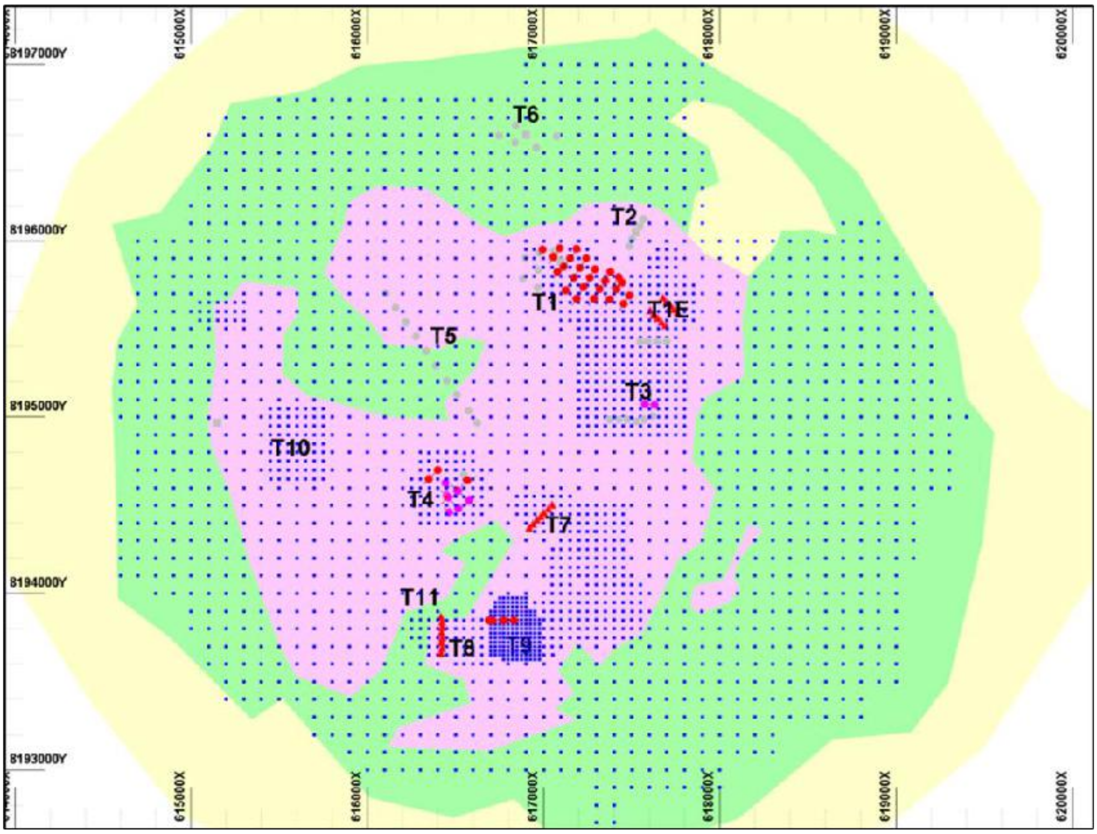
2023

From the start of 2023 to present, Altona's exploration work consists of:

- An additional 11 RC holes (total 752.39 m) were drilled between July and August 2023.
- The aim of these drill holes was to understand better the geometry of REE mineralisation at Target 4, and to follow up on the 2021 discovery at Target 3. At Target 3, drill hole MM110 intersected 2.735% TREO from surface to 30 m, open at depth, confirming that Target 3 warrants additional exploration.
- In-situ pXRF assays on cleaned outcrops in shallow trenches were also carried out at Target 3, Target 4, and newly identified Target 11. Results are currently being processed.

A plan view/ summary of Altona's 2022 and 2023 REE exploration campaigns is shown in Figure 4.24. Red dots are 2022 RC holes collars; red triangles are 2022 Short RC collars; purple dots are 2023 RC collars; grey dots and squares are 2021 holes collars (RC and DD respectively); blue dots are the soil sampling locations.

Figure 4.24 Altona 2022 and 2023 REE exploration summary



Source: Altona, 2021

4.5.3 Historical exploration target (Hattingh et al, 2022)

An Exploration Target estimate based on drilling samples assayed by pXRF was prepared in 2022 to guide future exploration (Hattingh et al, 2022; https://www.altonare.com/investors/reports/; https://www.altonare.com/investors/documents/). This was released publicly by Altona in 2022.

Exploration Targets were generated for both Target 1 and Target 4 using the defined block models constrained by the ground surface, distance to the nearest borehole, a depth below surface of 100 m in the case of Target 1 and between 80 m and 100 m below surface in the case of Target 4, and TREO grade shells: firstly a 0.5% TREO cut-off grade shell and secondly a 1.0% TREO cut-off grade shell for both Target 1 and Target 4. Exploration inventories have been reported for Targets 1 and 4 for both grade shells at a 1% TREO cut-off and a 2% TREO cut-off within the grade shells.

The total REO Exploration Target across Targets 1 and 4 is between 6.5 and 56.7 Mt at respective TREO grades of between 2.45% and 1.65% (Table 4.5).

Table 4.5 Historical Exploration Target range summary

Percent TREO cut-off applied within grade shell (%)		Tonnes (millions)		TREO%	
	Target	0.5% grade shell	1.0% grade shell	0.5% grade shell	1.0% grade shell
1.0%	Target 1	39.1	12.9	1.63	1.75
	Target 4	17.5	8.8	1.69	1.86
	Total	**56.7**	**21.6**	**1.65**	**1.79**
2.0%	Target 1	7.8	3.4	2.47	2.51
	Target 4	3.7	3.0	2.35	2.37
	Total	**11.5**	**6.5**	**2.43**	**2.45**

Snowden Optiro has been provided with the report but has not reviewed the veracity of the data or estimation method. Snowden Optiro is not responsible for the information contained within the 2022 Hattingh report. This is a historic exploration target and therefore has no bearing on the resources reported in this MRE.

4.6 Sampling, sample preparation, analysis and security

4.6.1 Drilling and logging

The drilling campaigns at Monte Muambe are shown in Table 4.6 below. These are split by year and drilling type.

Table 4.6 Summary of Monte Muambe drilling programmes

Programme	Number of holes	Metres drilled
2010-2012 Legacy RC	165	12,587
2021 DD	5	591
2021 RC	36	2,518
2022 RC	35	3,372
2022 short RC	26	594
2023 RC	11	789
TOTAL	**278**	**20,451**

Diamond drilling

Diamond boreholes were positioned by the Project Manager using a handheld GPS. All holes were vertical, except hole MM007 (Target 2) which was angled at 55°. Drill core was packed in 1 m length boxes with core blocks to mark depth and recovery. Core was delivered to the core yard at the project field camp.

Completed boreholes were surveyed using the company DGPS system, using the survey beacon at the field camp as a reference. The location of camp survey beacon was initially acquired by the base station

(GPS measurement) and was not tied to the local grid. Snowden Optiro is of the opinion that the internal survey co-ordinate and height system is adequate for estimation of Inferred and Indicated Mineral Resources. Further detail of the internal survey system is presented in the Internal Survey System section below.

Completed boreholes were cased with poly-vinyl chloride (PVC) piping for subsequent downhole surveying. Concrete marker blocks were placed on the borehole collars PVC casing with the borehole ID imprinted into the concrete.

Five DD core boreholes were drilled in the 2021 exploration programme with a total meterage of 590.7 m. A trailer mounted Atlas Copco CS14 rig was used. Diamond drill holes were started in PQ diameter (85 mm), with the diameter reduced to HQ (63.5 mm) and if necessary, NQ (47.6 mm) as dictated by ground conditions. Approximately 15.5% of diamond drilling was carried out in PQ diameter, 63.9% in HQ and 20.6% in NQ. Because of the disseminated nature of the mineralisation, it was not considered necessary to do core orientation.

The DD cores were checked against the driller's core blocks and recovery was recorded. The presence of cavities was recorded based on information provided by the driller and observations on the core. DD core recoveries varied from 17% to 100%, with an average of 83%. Short runs were used to maximize sample recovery when necessary. The entire length of each drill hole was logged by trained geologists. Lithology, mineralogy, colour, weathering, grain size, texture, fabric and alterations were logged using codes for each aspect as per Altona's standard operating procedure. Gamma spectrometer logging was carried out using a hand-held gamma spectrometer to do spot readings at 50 cm intervals of cores.

All DD core trays were photographed in standard conditions and white-balanced.

RC drilling

RC boreholes were positioned by the Project Manager using a handheld GPS. Borehole azimuth and inclination were planned by the Project Manager. Drill chips were collected via a cyclone in 1 m bags with each bag being weighed and the weight recorded. During drilling the drill string and cyclone were regularly flushed out with air. Splitting of 1 m samples was done at the drill site. A sample of chips was wet screened on site with the coarse chips being placed in chip boxes for logging purposes.

Completed boreholes were cased with PVC piping for follow up downhole surveying. Downhole surveys were conducted at 4 m intervals to the bottom of each borehole except where sidewall collapse prevented the full survey tool penetration. However, downhole survey data has not been incorporated into the company database records. Concrete marker blocks were placed on the borehole collars PVC casing with the borehole ID imprinted into the concrete.

Completed boreholes were surveyed using the Company's DGPS system, with the survey beacon at the field camp as a reference. The location of camp survey beacon was initially acquired by the base station (GPS measurement) and was not tied to the local grid. Snowden Optiro is of the opinion that the internal survey co-ordinate and height system is adequate for estimation of Inferred and Indicated Mineral Resources. Further detail of the internal survey system is presented under the Internal Survey System header below.

RC drilling was undertaken in both the 2021, 2022 and 2023 drilling programmes with 36 boreholes (2,518 m) in 2021, 31 boreholes (2,943 m) in 2022, and 11 boreholes in 2023 (752.39 m) with the latter boreholes not incorporated in the Mineral Resource estimate as laboratory assay results were not available at the reporting cut-off date.

The RC drill rigs were a truck mounted Smith Capital 14R6H with a 21 bar compressor (2021 and 2023 drilling campaigns) and a track mounted Hanjin Power 7000SD (2022 drilling campaign). The RC bit has a 4½ inch (114.3 mm) diameter.

The entire length of each drill hole was logged by trained geologists, using chip boxes. Lithology, colour, weathering, grain size, texture, fabric and alteration were logged using codes as per Altona's standard operating procedure.

The REE mineralisation is not visually evident related to lithology RC logging. From early 2002 onwards, the Hitachi X-MET8000 portable XRF (pXRF) analyser with a 50 kV anode design was used to assay Ce, La, Nd, Pr and Y.

For RC samples, a 50 g sub-sample was split from each 1 m sample using a 1-tier riffle splitter. Each sub-sample was split further and placed in an XRF capsule for assay. The pXRF was set up in bench top mode. Preparation and assay were done in standard conditions. The sum of the five aforementioned elements was calculated as oxide percent. Orientation, QC and comparisons with laboratory results show that this sum provides a reliable proxy of the actual TREO%. Accordingly, pXRF logging results were used to guide the day-to-day implementation of the drilling programme and to select mineralized samples (TREO>0.5%) to be sent to the laboratory for assay. The pXRF assay results were not used for the Mineral Resource estimation.

Lithology determinations on RC chips were supported by the preliminary pXRF assays done on site, with SiO_2 being used to distinguish fenite from carbonatite and from mixed lithologies, and MgO to distinguish two geochemically different suites of carbonatites.

Gamma spectrometer logging was carried out using a hand-held gamma spectrometer to do one reading in each RC cutting bag (RC samples). No geophysical tools were used to determine element grades. Geology logging was qualitative and pXRF logging was quantitative. RC chip trays were photographed in standard conditions and white-balanced.

Internal survey system

Altona procured a new Kolida K20S RTK/GNSS set and imported it in April 2022. The General Manager, who was already familiar with the use of dGPS systems, received equipment-specific training from the supplier in Kenya, and subsequently trained one of Monte Muambe’s field technicians to operate the survey equipment. Operation of the RTK survey equipment is covered by SOP 2023-02.

The RTK system has internal controls, the most important being horizontal standard deviation and vertical standard deviation (in essence the horizontal and vertical accuracy). Both should be below 10 mm, and preferably below 5 mm. This can only happen if the Base Station and the Rover are communicating correctly. These parameters are displayed during data recording and are saved in raw report files.

All borehole collars are systematically RTK-surveyed. All accessible legacy collars (meaning most of those existing) were RTK-surveyed by Altona. For each borehole collar, the actual position measured is that of a point at the top of the cement slab and on the edge of the protruding casing. This position is corrected in X and Y post-survey so that the collar position recorded in the collar file corresponds to the centre of the casing.

Drone-borne photogrammetry surveys were carried out by Altona on Targets 1, 4 and 9. The surveys were carried out between 28 September 2022 and 5 October 2022, at a time when the vegetation cover was at its minimum (height of the dry season, after grass fires swept through the work area).

The surveys were undertaken using a Mavic Air 2 drone, with the flight plans prepared on Drone Link software. Surveys were prepared and flown by a licensed drone pilot. The Mavic Air 2 is a non-RTK drone. Surveys were therefore georeferenced using visible hole collars (cement slabs with protruding casing), for which RTK-surveyed coordinates were available, as ground control points (GCP). Raw data for drone photogrammetry consists of folders of photographs each containing camera, flight and GPS information. These are archived in the project database, together with processed data and final products.

The data used for the MRE (Target 1 and Target4), however, was processed by Mr Kalumba Bwale of Snowden Optiro. Altona supplied raw data, raw photographs folders and a CSV table with GCPs to Snowden Optiro. The processing procedure involved loading the provided LAS files and generating a digital terrain model (DTM) from the points. This DTM was then smoothed using a function in DM Studio. The smoothed DTM was translated vertically by a nominal value above the loaded points (0.3 m in the document). All the original points below the smooth DTM were coded and kept and all the points above the smoothened DTM were deleted. This process was repeated as needed, until only the ground classified points remained. These points were not spatially manipulated in anyway. The resulting files contained fewer points and were filtered to a point density of 3 m.

4.6.2 Sampling

Diamond drill core sampling

Diamond core was cut with a diamond saw into quarter core (PQ and HQ) and half core for NQ core. All core was sampled using cut quarter or half core. All core was sampled at nominal 1 m lengths with adjustments made for lithological contacts. Actual sample lengths varied from 0.04 m to 3.36 m with an average of 0.62 m. Samples that were not competent enough to be split with the core cutter were bagged, homogenised and split using a riffler splitter.

Samples were collected by or under the guidance of the Project Manager with the cut half or quarter core placed in plastic sample bags with tag book ID markers stapled at the top of each sample bag. All samples were stored at the camp site sample store under the control of the Project Manager until collection by contract transporter, Bollore Logistics, which transported the samples to Intertek Genalysis laboratory near Johannesburg in South Africa.

Sampling was conducted in accordance with the project standard operating procedures (SOPs) which are in line with good industry practice.

RC sampling

The RC drill chips were first collected in 1 m plastic bags via a cyclone. These were weighed and then taken by the Company's sampling team to be split, using a 4:1 riffle splitter, to 6 kg. A second riffler splitter was then used to top up the 6 kg sample to the exact 6 kg weight. From 2022 onwards, a third splitter was then used to split a 50 g sample from the 6 kg sample. The Competent Person's representative observed the RC sampling procedure during his August 2023 site visit. The sampling team was both well trained and motivated with minimal spillage, and with the equipment cleaned between samples. The sampling was undertaken according to Altona's standard operating procedure which is in line with industry RC sampling standards.

In the 2021 RC drilling campaign, all 1 m intersections were sampled with a 6 kg sampled split onsite with a riffler splitter. These were then made into 3 m composites from the original split 1 m samples (6 kg). In 2022, after assay results were received the original 1 m samples were then made into 2 m composites based on samples with >0.5% TREO and these 2 m composites were resubmitted for analysis.

For the 2022 and 2023 RC drilling programmes, the pXRF was used to analyse each 1 m sample with the analytical results being used to select samples to be combined into 2 m composites. The pXRF sample for analysis was 50 g in weight which was split from each 1 m 6 kg. Only samples with a pXRF assay result of >0.5% TREO were selected for compositing.

The 2 m composite samples were made up using a riffler splitter at the camp sample store with the sample for analysis being 3 kg in weight with the 9 kg balance packed in a plastic bag for storage. The decision to make up 2 m composites was based on feedback from early modelling done as part of the 2022 Exploration Target estimation assignment (Hattingh et al, 2022).

Samples were collected by or under the guidance of the Project Manager and placed in plastic sample bags with tag book ID markers stapled at the top of each sample bag. All samples were stored at the camp site sample store under the control of the Project Manager until collection by contract transporter, Bollore Logistics, which transported the samples to Intertek Genalysis laboratory near Johannesburg in South Africa.

Sampling was conducted in accordance with Altona's SOPs which are in line with good industry practice.

Analysis

Samples for analysis were submitted to the independent analytical laboratory Intertek Genalysis (accreditation status ISO/IEC 17025); where sample preparation was undertaken at their laboratory at Bapsfontein (near Johannesburg) in South Africa as per the drying, crushing and pulverising procedures in Table 4.7. Pulps were then despatched by airfreight to Intertek Genalysis in Perth Australia.

Neither laboratory was inspected by the Competent Person, but Snowden Optiro is of the opinion that the accredited status of the laboratory together with QC sample results is adequate for the analytical results to be used in Mineral Resource estimation.

Table 4.7 Intertek Genalysis sample preparation codes

Soil, rotary air blast and reverse circulation samples	
Description	**Code**
Dry, pulverise up to 300 g	SP01
Dry, pulverise 300 g up to 1.2 kg	SP02
Additional wt>1.2 kg: dry, split, pulverise up to 1.2 kg, retain coarse reject	SP66
Drill core and rock	
Description	**Code**
Dry, crush ~10 mm, pulverise up to 300 g	SP11
Dry, crush ~10 mm, pulverise 300 g up to 1.2 kg	SP12
Additional wt>1.2 kg: dry, crush ~2 mm, split, pulverise up to 1.2 kg, retain coarse reject	SP67

At the Intertek Genalysis Perth laboratory, the procedures that were applied are discussed in Table 4.8:

- All samples were assayed for Al, Ba, Ca, Ce, Cr, Cs, Dy, Er, Eu, F, Fe, Ga, Gd, Hf, Ho, K, La, Lu, Mg, Mn, Na, Nb, Nd, P, Pr, Rb, S, Sc, Si, Sm, Sn, Sr, Ta, Tb, Th, Ti, Tm, U, V, W, Y, Yb, Zr, LOI.
- Major elements and some trace elements (including Ce and La) were assayed by Li Borate fusion followed by ICP-OES.
- Trace elements (including all REE, U, Th and Nb) were assayed by Li Borate Fusion followed by ICP-MS.
- Fluoride was assayed by alkaline fusion in a nickel crucible followed by specific ion electrode (SIE) analysis.

Table 4.8 Analysis types and included elements used at Intertek Genalysis Perth laboratory

LITH/203

WHOLE ROCK LI BORATE FUSION / ICP-OES

ELEMENT	RANGE %	FINISH	ELEMENT	RANGE %	FINISH	ELEMENT	RANGE %	FINISH
SiO2	0.01 - 100	OES	MgO	0.01 - 100	OES	Ba	0.005 - 5	OES
TiO2	0.01 - 100	OES	CaO	0.01 - 100	OES	Cr	0.002 - 5	OES
Al2O3	0.01 - 100	OES	Na2O	0.01 - 100	OES	S	0.01 - 30	OES
Fe2O3	0.01 - 100	OES	K2O	0.01 - 100	OES	LOI 1000°C	0.01 -100	
MnO	0.01 - 100	OES	P2O5	0.01 - 100	OES			

REE LI BORATE FUSION / ICP-MS

ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH
La	0.2 - 20%	MS	Eu	0.05 - 5%	MS	Er	0.05 - 5%	MS
Ce	0.5 - 30%	MS	Gd	0.05 - 5%	MS	Tm	0.05 - 1%	MS
Pr	0.05 - 10%	MS	Tb	0.02 - 2%	MS	Yb	0.05 - 5%	MS
Nd	0.1 - 20%	MS	Dy	0.05 - 5%	MS	Lu	0.02 - 1%	MS
Sm	0.05 - 10%	MS	Ho	0.02 - 2%	MS			

HFSE LI BORATE FUSION / ICP-MS

ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH
Hf	0.1 - 5%	MS	Ta	0.1 - 5%	MS	Y	0.5 - 50%	MS
Nb	0.1 - 5%	MS	Th	0.05 - 2%	MS	Zr	1 - 50%	MS

MINOR & TRACE LI BORATE FUSION / ICP-MS

ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH	ELEMENT	RANGE PPM	FINISH
Ba	0.5 - 5%	MS	Rb	0.1 - 5%	MS	U	0.05 - 30%	MS
Cr	20 - 5%	ICP- OES	Sc	10-5%	OES	V	10 - 5%	OES
Cs	0.05 - 1%	MS	Sn	1 - 5%	MS	W	1 - 5%	MS
Ga	0.1 - 1%	MS	Sr	0.2 - 20%	MS			

LITH/203 $60.70

FLUORIDE ANALYSIS BY SELECTIVE ION ELECTRODE

ELEMENT	DESCRIPTION	DL	CODE	PRICE
F	Carbonate fusion / SIE	50ppm	FC7/SIE	$26.00

4.6.3 Snowden Optiro comments

Although there was concern that the QC field sample insertion rate (3.6 to 3.7%) is less than the industry standard (5%), this was addressed by the comparative exercise between 3 m 2021 composites and 2 m composites. Snowden Optiro recommends that future QC insertion rates be brought into line with accepted industry practices.

Previously, anomalous QC results were not followed up immediately; therefore, Snowden Optiro recommends that during future analytical programmes the QC results should be assessed, and anomalous values or trends addressed immediately.

Notwithstanding the above concerns, Snowden Optiro is of the opinion that the sample analytical results are suitable for estimation of Mineral Resources to Inferred and Indicated categories.

4.7 Data verification/ QAQC

4.7.1 Data verification

Altona implemented a series of routine verifications to ensure the collection of reliable exploration data. All work was conducted by appropriately qualified personnel under the supervision of qualified geologists. Core logging, surveying and sampling were monitored by qualified geologists and verified routinely for consistency. Electronic data was captured and managed using MS Excel sheets stored in a Dropbox folder backed-up on an off-site server in real-time. Assay results were delivered electronically to Altona geologists by the primary laboratories and were examined for consistency and completeness.

The database was regularly backed up on a second Dropbox account, and on the Company's Sharepoint backup system. Scans of all paper documents (driller's daily reports, logs etc) are stored digitally in the database. Digital data were checked and validated against the original field sheets.

Significant drill hole intersections were verified by Altona's CEO. No twin DD-RC holes were drilled.

Prior to resource estimation, Snowden Optiro completed a phase of data validation on the digital sample data that included the following:

- Search for sample overlaps, duplicate or absent samples, checks for anomalous assay or survey results. No material issues were found by Snowden Optiro in the final database.
- Snowden Optiro noted that although not material, geological logging codes for the historical Globe drilling had not been converted to the Altona logging codes.

4.7.2 QAQC

QC samples were inserted into both DD core samples and RC samples in the form of CRMs (OREAS), Blanks (local quartzite) and field duplicates (quarter core and split RC chip composites). A selection of pulp samples was submitted to Nagrom in Perth, for external (umpire) analysis.

The internal QC samples and external sample analytical samples are summarised in Table 4.9. It is evident that the internal QC sample insertion at between 3.6 and 3.7%, expressed as % original samples, is lower than industry standards. The actual QC results will be discussed in detail to address this concern and, as well, the 3 m composite RC samples in 2021 are compared to the 2 m RC samples also from the 2021 drilling campaign in order to add to confidence in the analytical results.

Table 4.9 Altona's internal QC sample insertion / selection

	Batch	Samples	Blanks	Standards	Duplicates	Umpire
2021	MMM001	878	32	27	32	
	MMM002	635	23	23	24	37
2022	MMM004	1,232	44	45	45	72
	MMM005	215	8	8	8	13
	Total	**2,960**	**107**	**103**	**103**	**122**
	% Total		3.65	3.5%	3.7%	
	% MMM004 and MMM005		3.6%	3.7%	3.7%	5.9%

REEs include 17 different chemical elements, 15 of which are present at Monte Muambe. To simplify the QC review the elements used to assess this aspect were La and Ce, which are the most abundant, Nd, Pr, Dy and Tb, which are the most important in term of economic value, and Th and U, which are the most important potentially deleterious elements.

Blanks

A fine grained silica rock (Figure 4.25) was used as a blank material. Unfortunately, it is not free of REEs and U and Th. However, the levels of these elements present are too low to significantly affect assessment of contamination from a QC point of view.

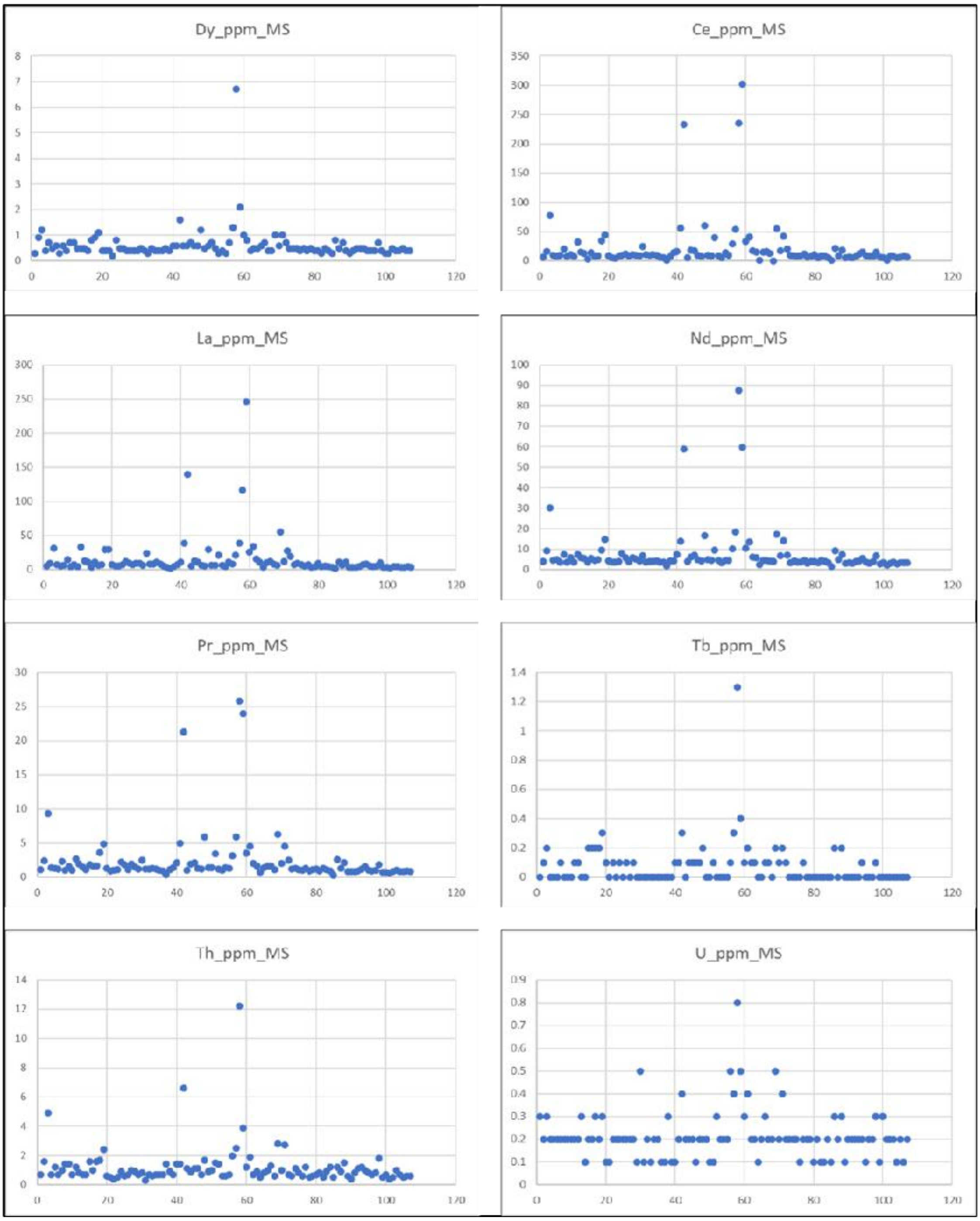
Table 4.10 gives a statistical summary of the QC blank sample results while Figure 4.26 shows the results in graphical format. It is evident that some results show higher levels of REE elements than expected which does indicate contamination, probably in sample preparation. However, the low level of the anomalous results does not, in the opinion of Snowden Optiro, affect the integrity of the original sample analytical results.

Figure 4.25 Silica rock used for QC blanks

Table 4.10 QC blank results statistics

Element	Mean	Standard Deviation	Min	Max
SiO_2 %	88.63	3.66	63.07	93.69
Al_2O_3 %	4.18	0.93	2.33	5.79
CaO %	0.53	0.25	0.24	2.57
Fe_2O_3 %	1.79	0.31	1.27	2.47
MgO %	0.51	0.10	0.29	0.71
MnO %	0.03	0.03	0.02	0.31
P_2O_5 %	0.02	0.03	0	0.27
Ce ppm	21	43	0	302
Dy ppm	0.62	0.65	0.20	6.70
La ppm	15	29	2	246
Nd ppm	8	11	2	87
Pr ppm	2.36	3.88	0.30	25.80
Tb ppm	0.07	0.15	0	1.30
Nb ppm	3.15	6.08	0	61.30
Th ppm	1.20	1.37	0.30	12.20
U ppm	0.21	0.11	0.10	0.80

Figure 4.26 QC blank sample assay results



Duplicates

Duplicates were inserted into the sample stream at the Project site. In the case of core quarter or half core was used for this purpose. In the case of RC samples duplicate samples were prepared during the preparation of composite 2 m samples. In all cases the duplicate samples were inserted into the sample stream adjacent to the original sample. No reject duplicates were selected to be inserted into the sample steam at Intertek Genalysis. In Snowden Optiro's opinion the core duplicates act more as an indication of short distance variation rather than true duplicates. The RC duplicate samples are deemed by Snowden Optiro to be true duplicates. The statistical and graphical analyses of the core and RC duplicates support the discussed differences in definition.

The duplicate QC insertion rates are low by industry standards. To compensate for this, to some degree, a comparison is made between the original 3 m assay results (not used for resource estimation) from the 2021 drilling programme and the 2 m samples (used for resource estimation) assay results.

Diamond core duplicates

DD field duplicates were inserted at a rate of 1 in 27.43 (3.65%) original samples. Scattergrams are shown in Figure 4.28 to Figure 4.30. R^2 varies from 0.94 to 0.98. Overall, it is the opinion of Snowden Optiro that these results are in line with expected short distance geochemical variation. To support this statement Figure 4.27 illustrates the chemical variability of REE bearing carbonatite.

Figure 4.27 XRF map showing Ce content variability in a core sample

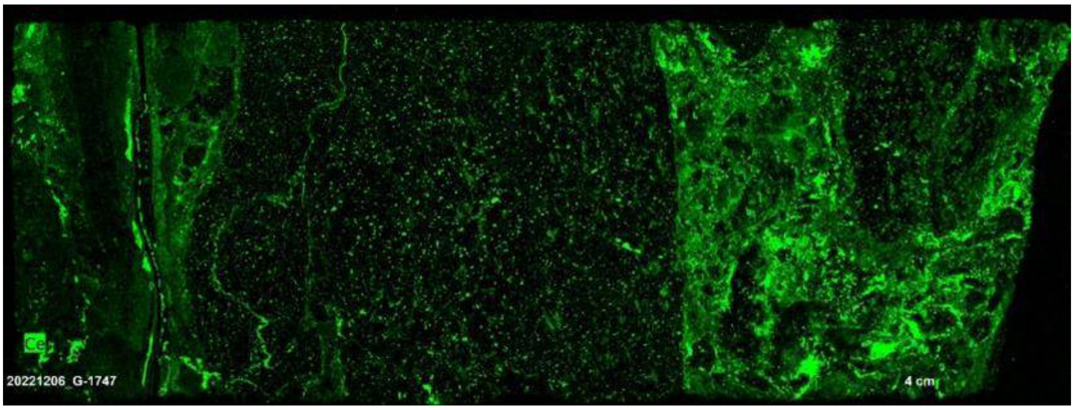


Figure 4.28 DD duplicate vs original sample assay results (Left Ce, right Dy)

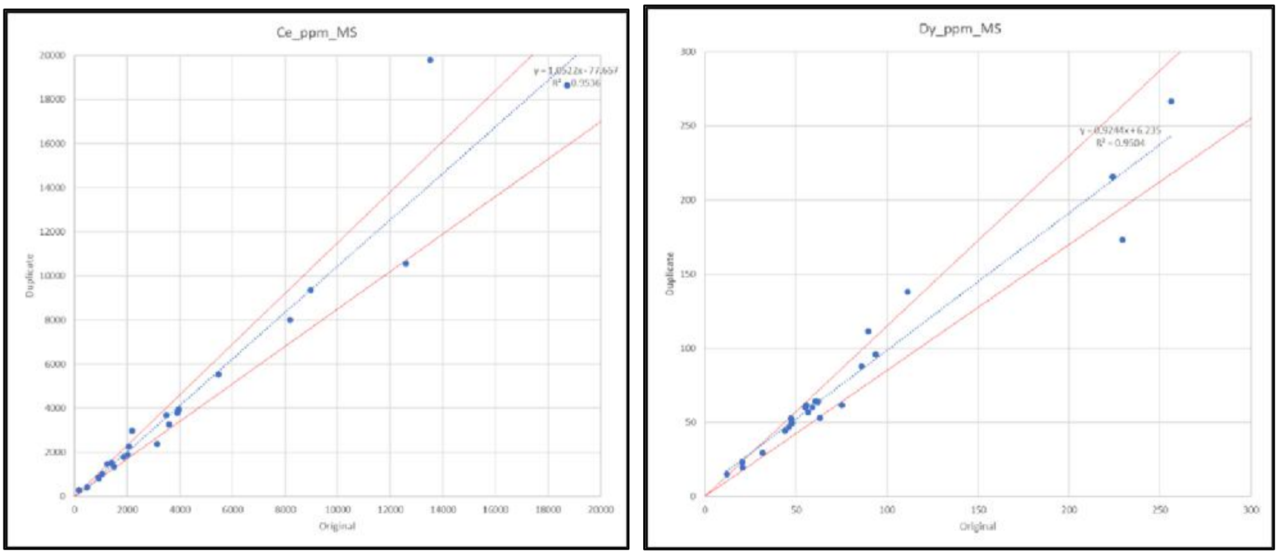


Figure 4.29 DD duplicate vs original sample assay results (Left La, right Pr)

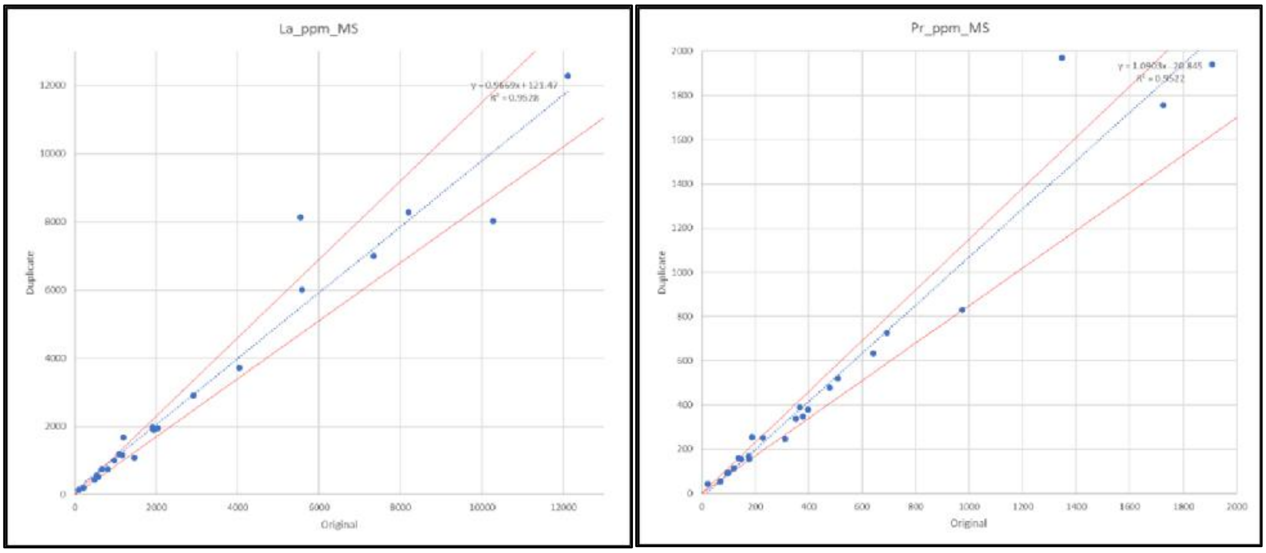
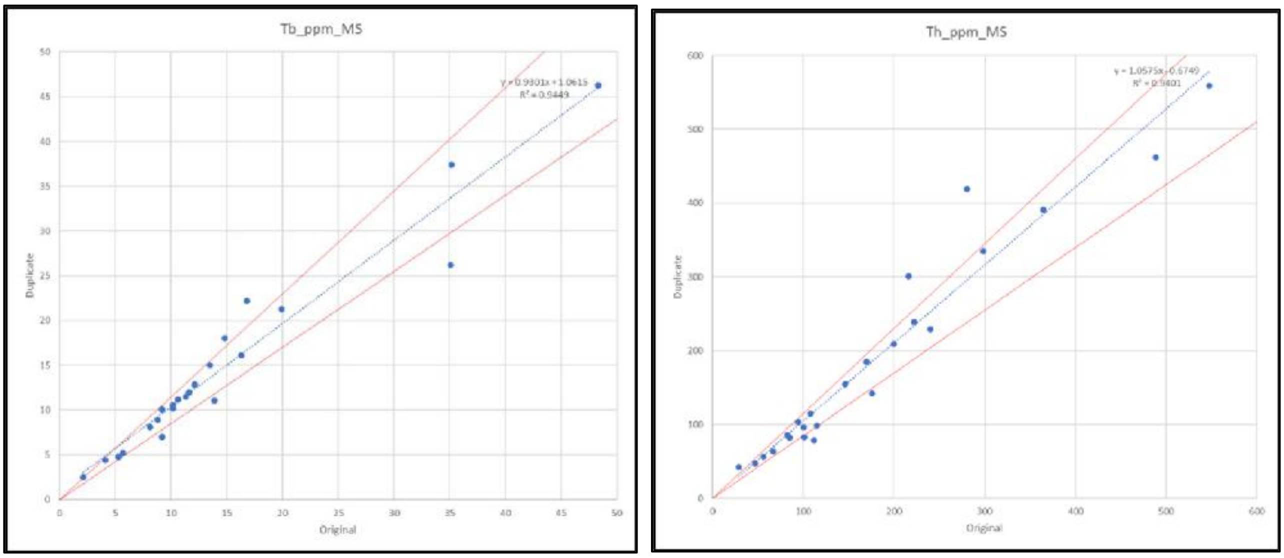


Figure 4.30 DD duplicate vs original sample assay results (Left Tb, right Th)



RC duplicates

RC field duplicates were produced by splitting a second sample from the selected 1 m RC cuttings bag using the same procedure as the original sample. The duplicates were inserted in the sample stream adjacent to the original samples in a ratio of 1 in 27.43 original samples.

Scattergram graphs for selected elements are shown in Figure 4.31 and Figure 4.33. All results, except for one U result, are well aligned with R^2 between 0.97 to 0.98. The singular U anomaly is not deemed significant and in Snowden Optiro's opinion the results show excellent correlation.

Figure 4.31 RC Ce duplicate vs original assays (Left Ce, right Dy)

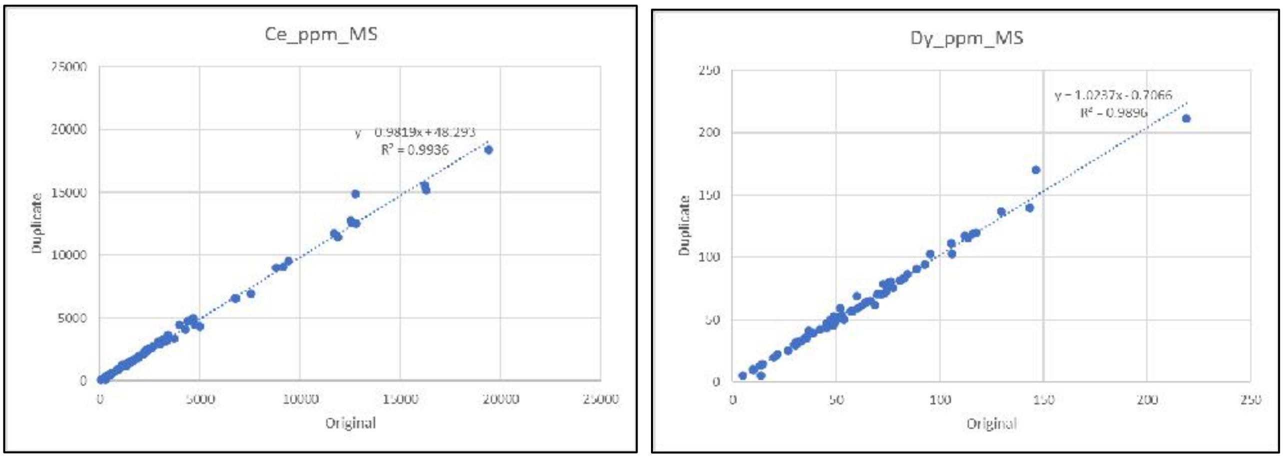


Figure 4.32 RC La duplicate vs original assays (Left La, right Pr)

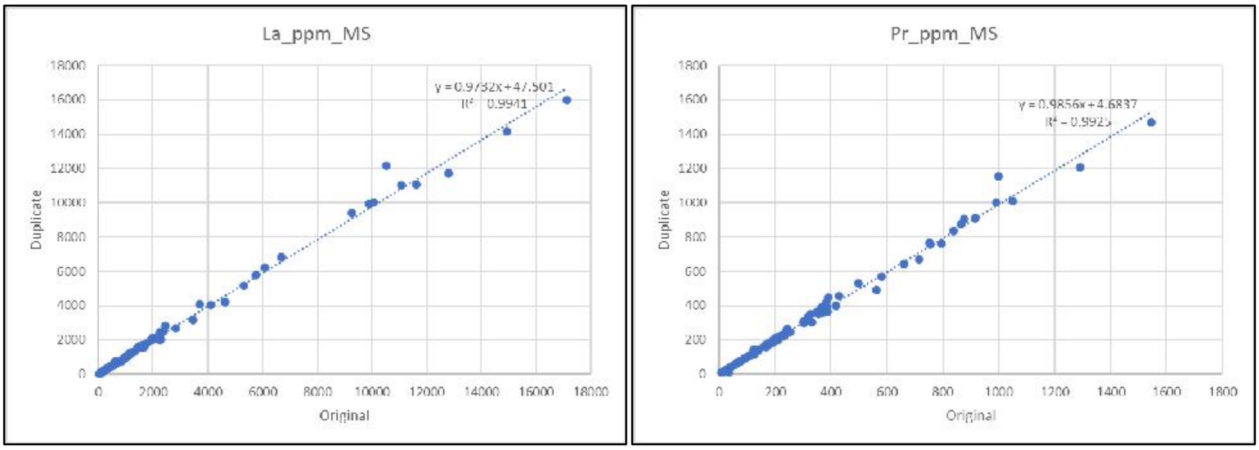
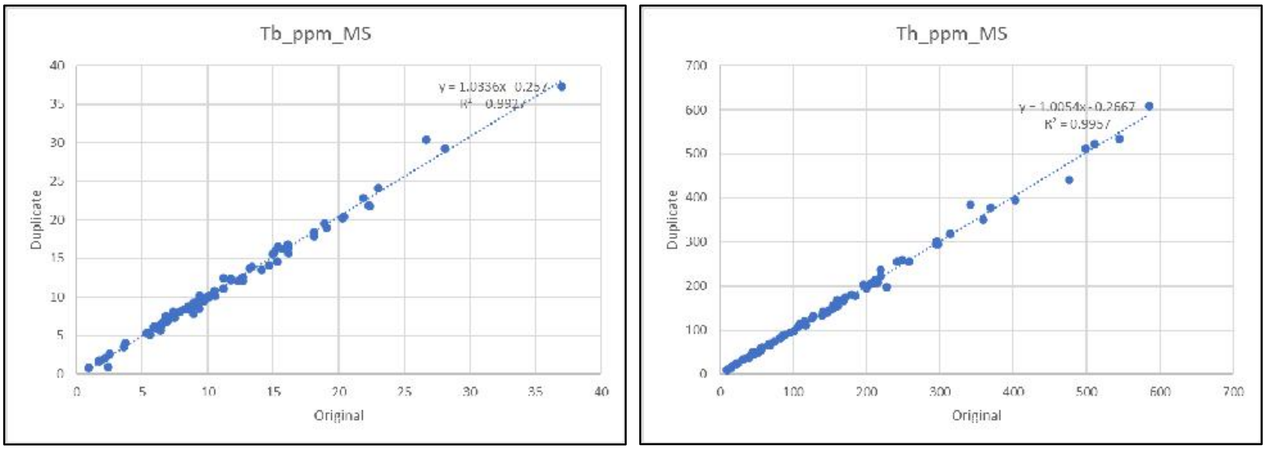


Figure 4.33 RC Tb duplicates vs original assays (Left Tb, right Th)



Comparison between 3 m and 2 m 2021 composites

RC drilling samples from the 2021 drilling campaign were initially assayed as 3 m composites to identify which intercepts justified more detailed (shorter intervals) assays and to save on assaying costs. Samples having TREO>0.5% were subsequently re-assayed as 2 m composites, together with the 2 m composites from the 2022 drilling campaign.

The two datasets therefore provide an opportunity to compare the original laboratory TREO results with the re-assay TREO results.

To prepare data for this comparison:

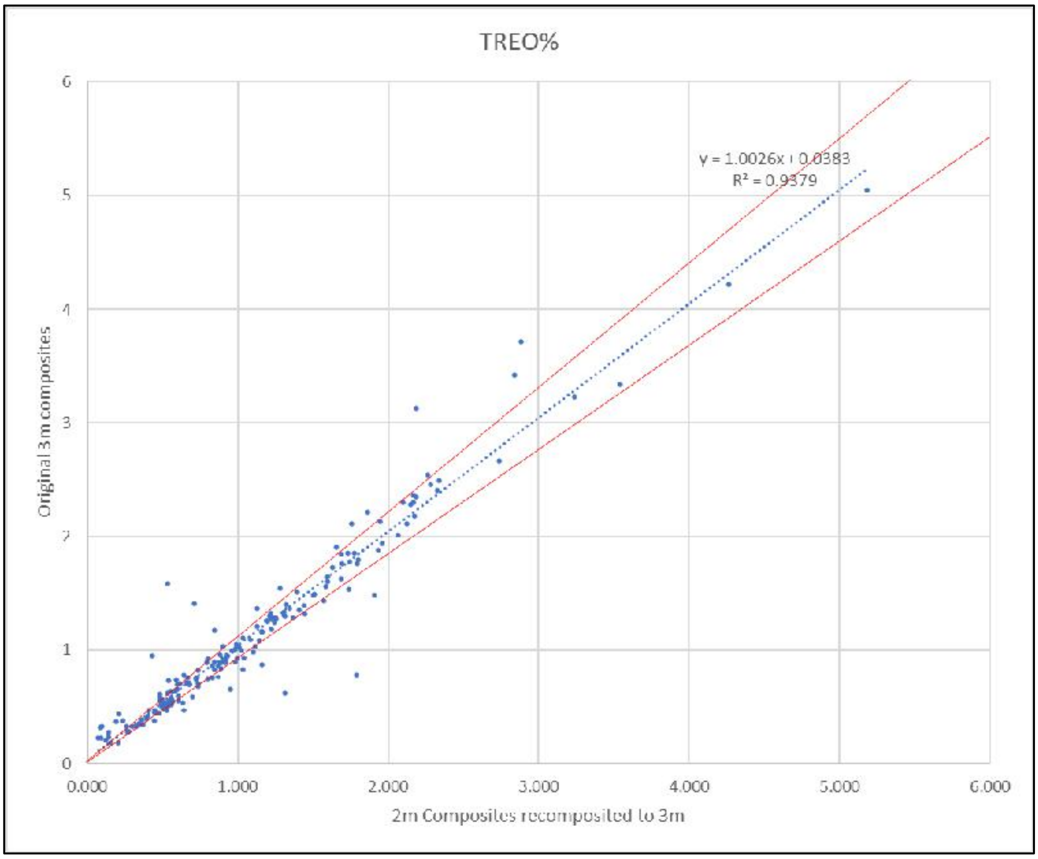
- The 2 m composites log file was re-composited using Micromine software so that From and To matched those of the 3 m composites log file.
- Both datasets were merged into the same MS Excel table.
- Entries for which results were available for only one of the two datasets were removed. This included all the samples which were not re-assayed.
- The correspondence of the Hole ID, From and To for each entry was checked and confirmed.
- Scattergrams for various elements were prepared.

The exercise has its limitations, because the preparation method for the 3 m composites in 2021 involved preparing each of the three 1 m sample so that their weight was within 200 g of each other (for 1 m sample weights ranging from 2 to 3 kg), while the preparation method for the 2 m composites in 2021 involved weighting each of the two 1 m sample so that its weight was identical to that of the other.

As described above, the 2 m composite database was re-composited to 3 m. In a situation where grade variations often occur at a short distance (from one 1 m sample to the other), discrepancies between the two datasets are more likely to be due to the compositing method rather than to a problem with the laboratory's operation.

In the opinion of Snowden Optiro, the comparison between the two samples sets related to TREO values supports the validity of the overall 2 m analytical results.

Figure 4.34 Comparison between 3 m and 2 m composite sample %TREO assay results



External (umpire) laboratory sample pulp duplicates

A total of 122 pulps from batches MMM002 (DD 2021), MMM004 (RC 2021 2 m composites) and MMM005 (RC 2022 2 m composites) was retrieved by Intertek under instruction from the Altona General Manager and sent to NAGROM laboratory in Perth for external (umpire) assays. This corresponds to a rate of 1 umpire sample to 17 original samples (5.88%).

A statistical analysis of the two sets of sample assay results is shown in Table 4.11. Overall, the two sets of results show good comparative values with the exception of dysprosium which was followed up with Intertek and NAGROM. Both laboratories are of the opinion that the difference in dysprosium results is due to minor differences in analytical procedure.

Table 4.11 Statistical summary of comparison between Intertek and NAGROM analyses

	Trendline Equation	R^2	Outliers -15% +15%
La	y = 0.9826x + 53.636	0.991	0
Ce	y = 0.9462x + 128.98	0.991	1
Nd	y = 0.9762x + 43.625	0.987	0
Pr	y = 0.9551x + 15.337	0.988	0
Dy	y = 1.0900x – 1.3264	0.986	8
Tb	y = 0.9828x + 0.0434	0.984	0
Th	y = 1.0241x + 2.1929	0.997	1

	Trendline Equation	R^2	Outliers -15% +15%
U	y = 0.9906x + 0.4922	0.989	4

CRM QC results

OREAS CRMs were used as standards for QC insertion into both the 2021 and 2022 drill sampling programmes:

- OREAS 46 and OREAS 47 (glacial tills, low rare earth levels). Beside their role as standards, these CRMs also act as blanks.
- OREAS 460, 461, 462 and 463 (REE-Nb carbonatite ore, Mount Weld, diluted with barren siltstone from the Melbourne area).

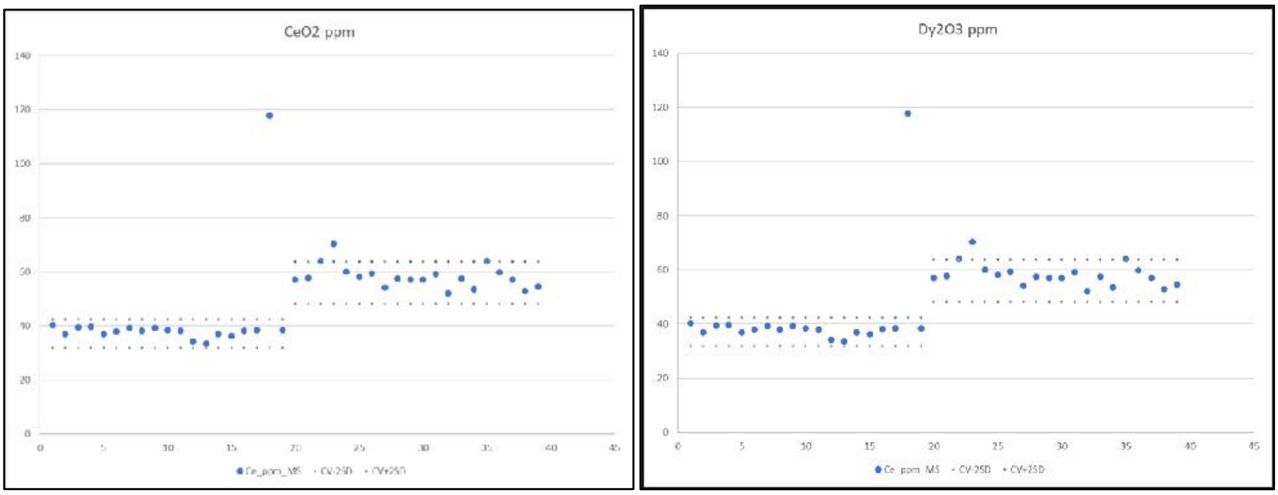
Standards were inserted at rate of 1 to 28.7 (3.48%) samples in average (1 to 27.3 (3.66%) for batches MMM004 and MMM005).

The control charts for selected REE elements are shown below, the first two for OREAS 46 and 47 with low REE values followed by combined control charts for OREAS 460, 461, 462 and 463 with progressively increasing REE contents.

As detailed under the relevant control chart there was only one CRM result that was significantly outside the control limits (certified value ± 2 x standard deviations). The remaining CRM results were mainly within the control limits and those outside being within 3 x standard deviations.

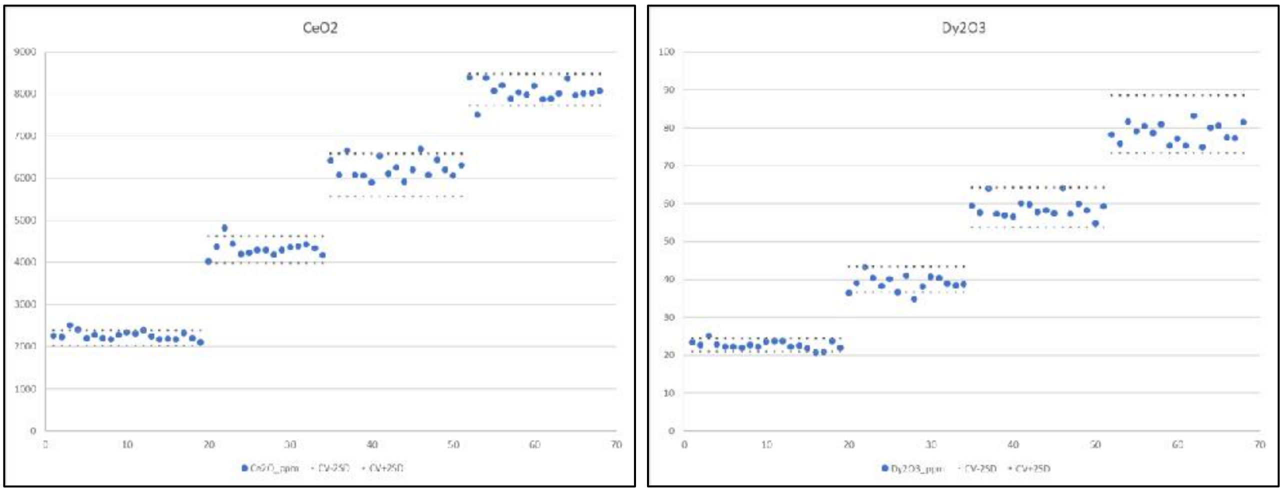
One OREAS 46 CRM sample (Z0687) result was significantly outside the control chart range (Certified Value ± 2 x standard deviations). The sample has been re-assayed by Intertek on request of the Company, with the new result falling within the control chart range (36.5 ppm Ce).

Figure 4.35 Control chart for CRMs OREAS 46 (left) and 47 (right)



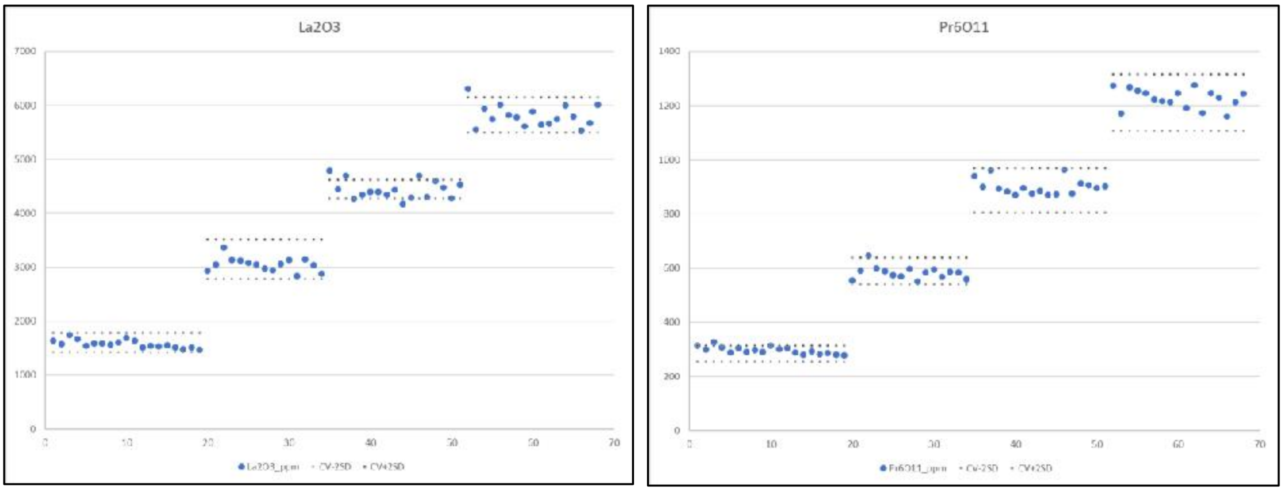
Note: CeO_2 left graph, Dy_2O_3 right graph.

Figure 4.36 Control charts for CRMs OREAS 460, 461, 462 & 463 (left to right)



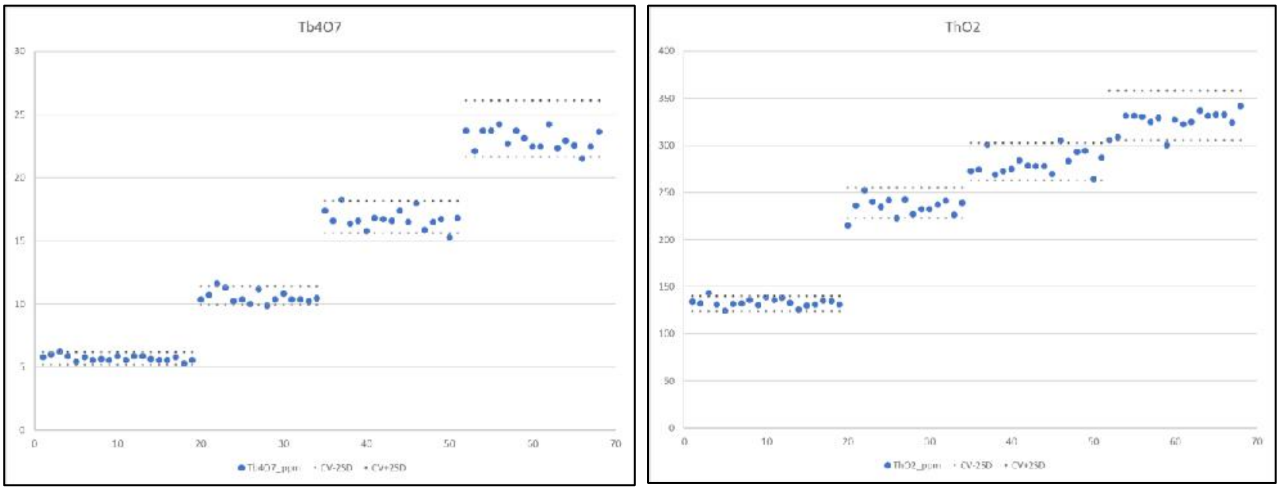
Note: CeO_2 left graph, Dy_2O_3 right graph.

Figure 4.37 Control Charts for CRMs OREAS 460 461 462 & 463 (left to right)



Note: La_2O_3 left graph, Pr_6O_{11} right graph.

Figure 4.38 Control charts for CRMs OREAS 460, 461, 462 & 463 (left to right)



Note: Tb_4O_7 left graph, ThO_2 right graph.

4.7.3 Snowden Optiro comments

Although there was concern that the QC CRM, Blanks and duplicate field sample insertion rate (3.6 to 3.7%) is less than the industry standard (5%), this was addressed to a degree by the comparative exercise between 3 m 2021 composites and 2 m composites. Snowden Optiro recommends that future QC insertion rates be brought into line with accepted industry practices.

External laboratory pulp analyses were conducted on a selection of pulps representing 5.9% of original pulp samples. Results were mainly in line with original analytical results with a minor difference in dysprosium results which was not deemed significant regarding resource estimation.

Previously, anomalous QC results were not followed up immediately; therefore, Snowden Optiro recommends that during future analytical programmes the QC results should be assessed, and anomalous values or trends addressed immediately. Anomalies that did occur with blank QC samples were not deemed to be significant regards resource estimation.

Notwithstanding the above concerns, Snowden Optiro is of the opinion that the sample analytical results are suitable for estimation of Mineral Resources to Inferred and Indicated categories.

4.8 Mineral Resource estimate

4.8.1 Data verification

The available data was supplied as a series of comma-separated variable (CSV) collar, lithology logging, assay, and sample/core recovery data, with the files grouped by year and drill hole method (2021 RC, 2021 DDH, 2022 RC and 2023 RC). In addition, the diamond drill hole data had down hole survey and density data.

Except for the assay data, each type of data (collar, logging and recovery data) was compiled into a single CSV file and imported into Datamine before any further processing. The assay data was imported into Datamine as individual tables to then be combined into a single assay file.

The surveyed collar coordinates surveyed using real-time kinematic - global positioning tool (RTK-GPS) which were supplied in a separate file. There was a total of eight drill holes which did not have RTK-GPS coordinates supplied, seven from the previous Globe data set and one from the Altona data set (MM092 which was abandoned at a depth of 6 m. These final RTK-GPS coordinate data were imported into Datamine and then merged with the collar data to create a single collar file with the correct coordinates, which were then validated against the available photogrammetry topography. No material discrepancies were identified. There are 26 Altona drill holes which are prefixed SRC*, which are the prefix for the short RC drill holes. Table 4.12 below shows the survey type split by drilling method.

Only the diamond drilling had downhole survey data, which was completed using a north-seeking gyro. All of the RC drilling used the design dip and bearing from the collar files were used for the downhole desurveying of the drilling. All of the RC drilling had the design bearing and azimuth applied to the collar and end of hole locations for desurveying of the drilling in three dimensional space.

Table 4.12 Monte Muambe collar survey type by drilling method

Company	Hole type	Collar survey method	No. holes	Metres drilled (m)
Globe	RC	Uncertain	7	498
		RTK-GPS	158	12,089
	Total Globe RC		**165**	**12,587**
Altona	RC	Uncertain	1	6
		RTK-GPS	82	6,520
	SRC*	RTK-GPS	26	594
	Total Altona RC		**109**	**7,120**
	Total Altona DDH		**5**	**591**
	Total Altona RC + DDH		**114**	**7,710**
Total drilling			**279**	**20,297**

Assay data

A total of 10,347 sample and assay records were available for the 2023 MRE update (Table 4.13). Both the Altona and historical Globe assay data contained multiple records with an assay value of '0' and/or comments indicating that these intervals were not sampled. In addition, the last tranche of handheld XRF data had duplicated intervals. All non-assayed and duplicated assays were extracted to a separate file for checking and then removed prior to desurveying. All of the Altona analytical data for the MRE were assayed by Intertek Australia, Maddington premises in Perth Western Australia. Analysis was by induced coupled plasma, either optical emission spectroscopy (ICP-OES) or mass spectroscopy (ICP-MS), as outlined in Table 4.14.

Table 4.13 Available assay data by target, source and hole type

Target	Source	Hole type	Handheld XRF		Laboratory assay	
			Nos	Length (m)	Nos	Length (m)
1	Globe	RC	-	-	653	2,060
	Altona	RC	246	248	1,136	2,210
		DDH	-	-	225	154
		Sub-total	**246**	**248**	**1,361**	**2,364**
	Total T1		**246**	**248**	**2,014**	**4,424**
4	Globe	RC	-	-	295	1,147
	Altona	RC	1,282	1,279	256	508
		DDH	-	-	-	-
		Sub-total	**1,282**	**1,279**	**256**	**508**
	Total T4		**1,282**	**1,279**	**551**	**1,655**
6	Globe	RC	-	-	246	838
	Altona	RC	-	-	55	106
		DDH	-	-	252	150
		Sub-total	**-**	**-**	**307**	**256**
	Total T6		**-**	**-**	**553**	**1,094**
Other	Globe	RC	-	-	3,448	5,584
	Altona	RC	-	-	-	-
		DDH	-	-	167	100
		Sub-total	**-**	**-**	**167**	**100**
	Total Oher		**-**	**-**	**3,615**	**5,684**
Total	Globe	RC	-	-	4,642	9,629
	Altona	RC	1,528	1,527	1,447	2,824
		DDH	-	-	644	404
		Sub-total	**1,528**	**1,527**	**2,091**	**3,227**
	Total		**1,528**	**1,527**	**6,733**	**12,856**
Not sampled '0' values**	Globe	RC	-	-	1,818	2,958
	Altona	RC	-	-	201	378
		DDH	-	-	-	-
		Sub-total	**-**	**-**	**2,019**	**3,336**
Duplicate records**	**Altona**	**Total**	**67**	**67**	**-**	**-**

*Note: ** removed prior to desurveying; RC – reverse circulation; DDH – Diamond drill hole*

Table 4.14 Analytical methods

Analytical method	Variables
ICP-OES	Al_2O_3 (0.01 %), Ba (50 ppm), CaO (0.01 %), Ce (0.5 ppm), Cr (20 ppm), Fe_2O_3 (0.01 %), K_2O (0.01 ppm), MgO (0.01 %), MnO (0.01 %), Na2O (0.01 %), P_2O_5 (0.01 %), S (0.01 %), Sc (10 ppm), SiO_2 (0.01 %), TiO_2 (0.01 %), V (10 ppm)
ICP-MS	Ba (0.5 ppm), Cr (20 ppm), Cs (0.1 ppm), Dy (0.1 ppm), Er (0.1 ppm), Eu (0.1 ppm), Ga (0.1 ppm), Gd (0.1 ppm), Hf (0.1 ppm), Ho (0.1 ppm), La (0.2 ppm), Lu (0.1 ppm), Nb (0.1 ppm), Nd (0.1 ppm), Pr (0.1 ppm), Rb (0.1 ppm), Sm (0.1 ppm), Sn (1 ppm), Sr, 0.2 (ppm), Ta (0.1 ppm), Tb (0.1 ppm), Th (0.1 ppm), Tm (0.1 ppm), U (0.1 ppm), W (1 ppm), Y (0.5 ppm), Yb (0.1 ppm), Zr (1 ppm)
Selective ion electrode (SIE)	F (50 ppm)
Thermo-gravimetric @ 1,000)	LOI-1000 (0.01 %)

The TREO values were supplied by Altona as part of the assay data, represents the sum of the rare earth oxide values (CeO_2, Dy_2O_3, Er_2O_3, Eu_2O_3, Gd_2O_3, Ho_2O_3, La_2O_3, Lu_2O_3, Nd_2O_3, Pr_6O_{11}, Sm_2O_3, Tm_2O_3, Tb_4O_7, YwO_3 and Tb_2O_3). Snowden Optiro checked the TREO values.

The sample REE and REO values were converted to parts per million (multiplied by 10,000) to ensure the fidelity of the assay grade during estimation and the various major elemental oxide assays (Fe_2O_3, SiO_2, TiO_2, Al_2O_3, etc) were kept as percent values.

As the Globe data had multiple element/oxide value combinations, all element and stoichiometric oxide values were calculated if absent. For the major element/oxide values the conversion factors were sourced from the AusIMM Monograph 9, Table 4.3. The REE /oxide conversion, the factors were sourced from the James Cook University Advanced Analytical Centre (JCU-AAC) website Advanced Analytical Centre - Element-to-stoichiometric oxide conversion factors - JCU Australia, with the key conversion factors presented in Table 4.15.

Table 4.15 Element/oxide conversion factors

Element	Oxide	Factor	Element	Oxide	Factor
Ce	Ce_2O_3	1.1713	Pr	Pr_2O_3	1.1703
	CeO_2	1.2284		Pr_6O_{11}	1.2082
Dy	Dy_2O_3	1.1477	Sm	Sm_2O_3	1.1596
Er	Er_2O_3	1.1435	Tb	Tb_2O_3	1.151
Eu	Eu_2O_3	1.1579		Tb_4O_7	1.1762
Gd	Gd_2O_3	1.1526	Tm	Tm_2O_3	1.1421
Ho	Ho_2O_3	1.1455	Yb	Yb_2O_3	1.1387
La	La_2O_3	1.1728	Nb	Nb_2O5	1.4305
Lu	Lu_2O_3	1.1371	Sc	Sc_2O_3	1.5338
Nd	Nd_2O_3	1.1664	Yb	Y_2O_3	1.2699

It was noted that for barium oxide (BaO) there was a single value in the supplied data of >5 (drill hole MM055, 34 - 36 m, sample B8790) – this value was re-set to a value of 5.75 to reflect a grade greater than 5.00 (the detection limit) and greater than the highest returned assay of 5.58 but not excessively high. The final imported assay dataset was then cross-checked against the available assay certificates where the only discrepancies identified were related to rounding of values during oxide-elemental conversions.

Lithological data

There are effectively 4 sources of geological logging/rock descriptions in the available data:

- Globe Metals logging (‘LITH 1’ and ‘Lith 2’).
- Altona geological logging (‘LITHO’).
- Altona REE logging derived from the available geochemistry (‘REV_L’).
- Altona REE logging derived of the available Globe available geochemistry (file ‘MMDB_LITHO_FROM_ASSAYS.XLS ‘, field ‘LITHO’).

As a function of the different file structures, it was noted that some data captured the presence of cavities/voids in the ‘LITHOLOGY’ fields, some by way of a VOID field, and others in the ‘COMMENT’ field. If either the LITHOLOGY or COMMENT fields noted the presence of a void a value of ‘-99’ was added to the ‘VOID’ field to denote the presence of a potential cavity not identified during logging, that could then be reviewed and used as required. In reviewing available core photographs (Figure 4.39), it was noted that the voids were millimetre to centimetre scale features.

Density data

Density determinations was only available for the diamond drilling and determined using a calliper method (weight of an interval / nominal core volume). Due to the limited number of density determinations available and the risk of an errant determination unduly influencing the outcome, only 416 of the 454 available density determinations were deemed valid, and 38 records were viewed as being unreliable:

- MM040 had six replicated samples – single duplicates for SB203, SB201 and SB202, and S204/ SB204 replicated three times.
- Thirty-one samples which had the hole diameter estimated because the recovered material was highly weathered or used a nominal core diameter.
- A single sample (MM001, 48.32 to 48.57 m returned a density value of 4.74 t/m^3 which as noted as dubious, as the near identical looking material either side of this interval had density values of 2.63 and 2.65 t/m^3.

Figure 4.39 MM040 box 36 106.35 to 110.95 m showing typical void presentation

Data preparation

All available data was imported into Datamine Studio RM Pro (v1.12.113.0) for subsequent desurveying and preparation. The collar coordinate data was updated with the final RTK-GPS coordinates and validated spatially.

For the geological logging data, there were nine drill holes with maximum interval depths less than either the collar drill hole depth or the maximum assay depth (e,g MM051 was drilled to 84.5 m depth, the assay data recorded a maximum depth of 84.5 m but the maximum depth for the geological logging was recorded as 85.0 m). The discrepancies were all less than 0.5 m, with the geological logging interval being rounded to the nearest metre drilled. After consultation with Altona and a review of the available data, for these nine drill hole, the end of hole interval depth for the geological logging data was re-set to the maximum depth recorded in the collar and assay tables.

The available assay data was captured in four groups of files:

- Altona diamond assay data.
- Altona RC drilling campaigns were assayed using ICP at Intertek Laboratories in Perth.
- Altona RC drilling completed in August 2023, assayed with handheld XRF data.
- Assay data from Globe Metals and Mining Ltd.

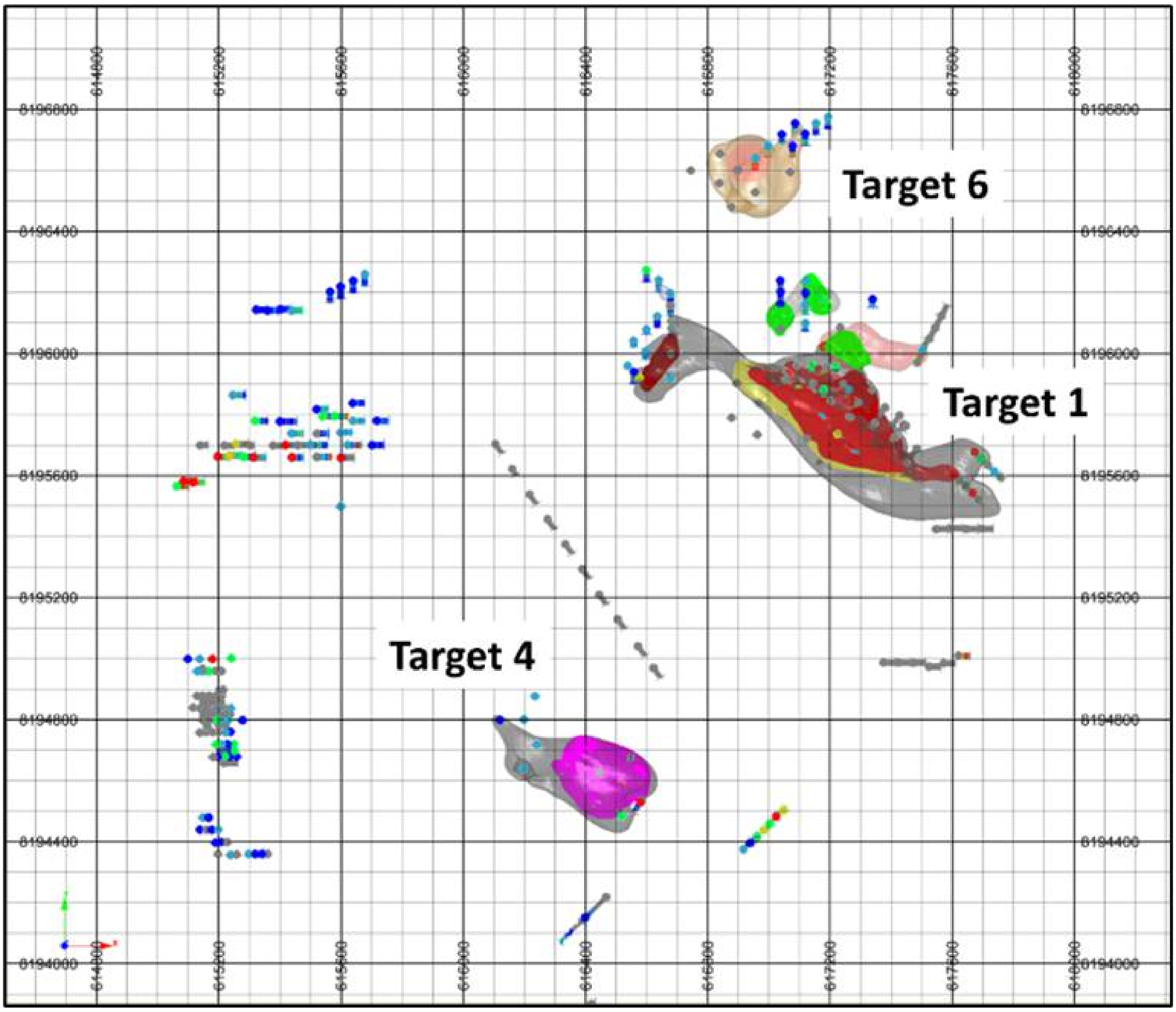
The Altona and Globe geology data employed different rock and weathering code legends and although there is broad correlation between the two, there are also some minor differences. Currently there is no translation table available between the two data sources. The weathering code for the Globe data use identical weathering codes as the Altona legend (fresh – FR and soil – SO as well as sap rock SR which matched spatially the Altona slightly weathered). However, after a spatial review of the weathering codes, the Globe logging codes SP (saprolite) and TR (transitional) had no identifiable translation to the Altona weathering codes.

It was noted that some of the drill hole recovery data had the drilled interval metres equal to the cavity metres (implying 0% total recovery), but the recorded recovery was calculated as 100%. To fully assess the core recovery, the total recovered core (TRC%) was calculated excluding any cavity length. Note that the diamond core is recorded as length of recovered core, but the RC recovery is recorded as weight of sample.

4.8.2 Mineral Resource estimation

Snowden Optiro was requested to prepare interpretations in support of a Mineral Resource estimate (MRE) for the Monte Muambe mineralisation for Targets 1 and 4 (Figure 4.40). As a function of the proximity of Target 1 to Target 6, interpretations were also completed for Target 6. Interpretation and estimation modelling at Target 6 was prepared with the aim of providing a possible exploration opportunity and scenario planning information only, not for reporting as a Mineral Resource.

Figure 4.40 Plan view mineral inventory areas



Exploratory data analysis

Initial exploratory data analysis was undertaken comparing the global statistics for the separate target areas (combined Target 1 and Target 4). Target 1 has substantially more data compared to target 4 and the analysis was primarily conducted using the Target 1 data, and tested for Target 4, which exhibited similar patterns. For the exploratory data analysis, length weighted statistics were prepared using Snowden Supervisor software.

Box and whisker plots were prepared examining the logged rock and weathering codes against the TREO grades. It was observed that lower TREO grades were associated with material logged as fresh material compared to the more weathered material (Figure 4.41, Figure 4.42 and Figure 4.43).

Figure 4.41 TREO% box and whisker plot global Monte Muambe by logged weathering

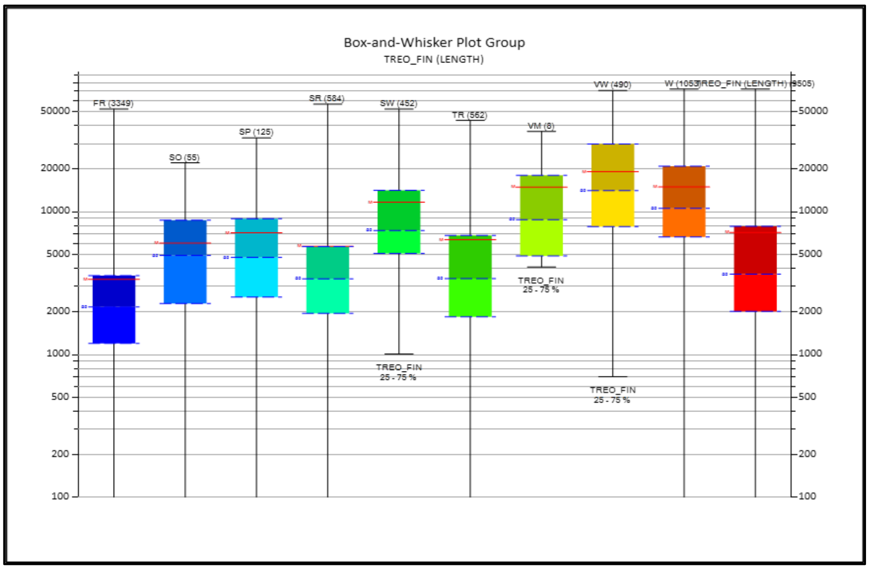


Figure 4.42 TREO% box and whisker plot Target 1 by logged weathering

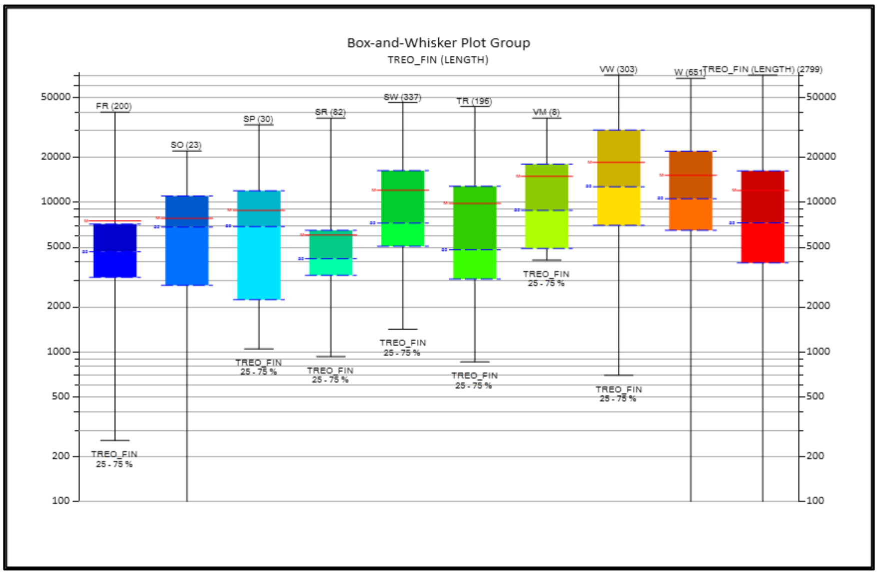
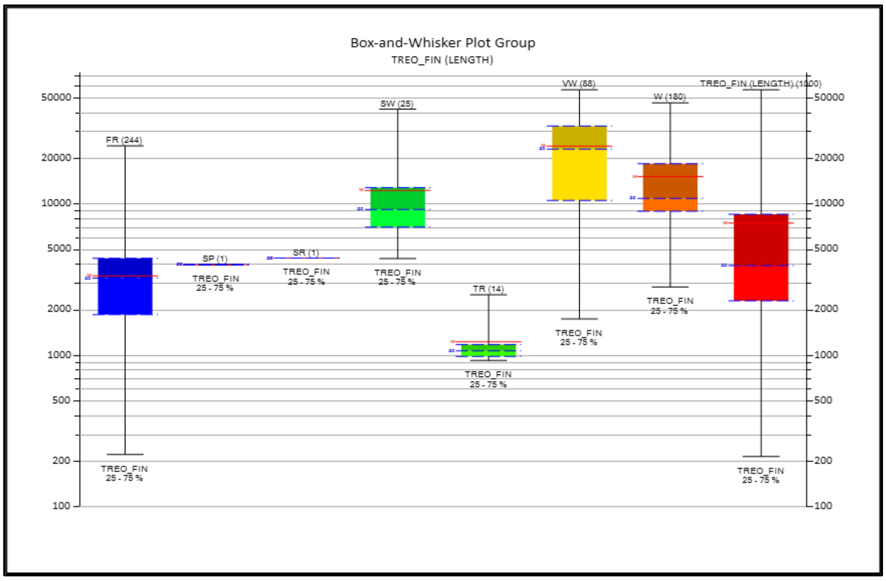


Figure 4.43 TREO% box and whisker plot Target 4 by logged weathering



Box and whisker plots were also prepared by logged lithology (Figure 4.44), where it was noted that:

- The undifferentiated and magnesium rich lithologies (CAR, CMG and MMG) had an overall higher average TREO grade compared to the other rock types.
- Globally, material logged as fenite had a lower average grade of 0.47% TREO compared to the carbonatite lithologies, but some were still mineralised (Figure 4.45). For the finites, 30% of the data exceed 0.5% TREO, and 10% of the fenite have grades exceeding 1.0% TREO.

Figure 4.44 TREO% box and whisker plot by logged lithology global Monte Muambe

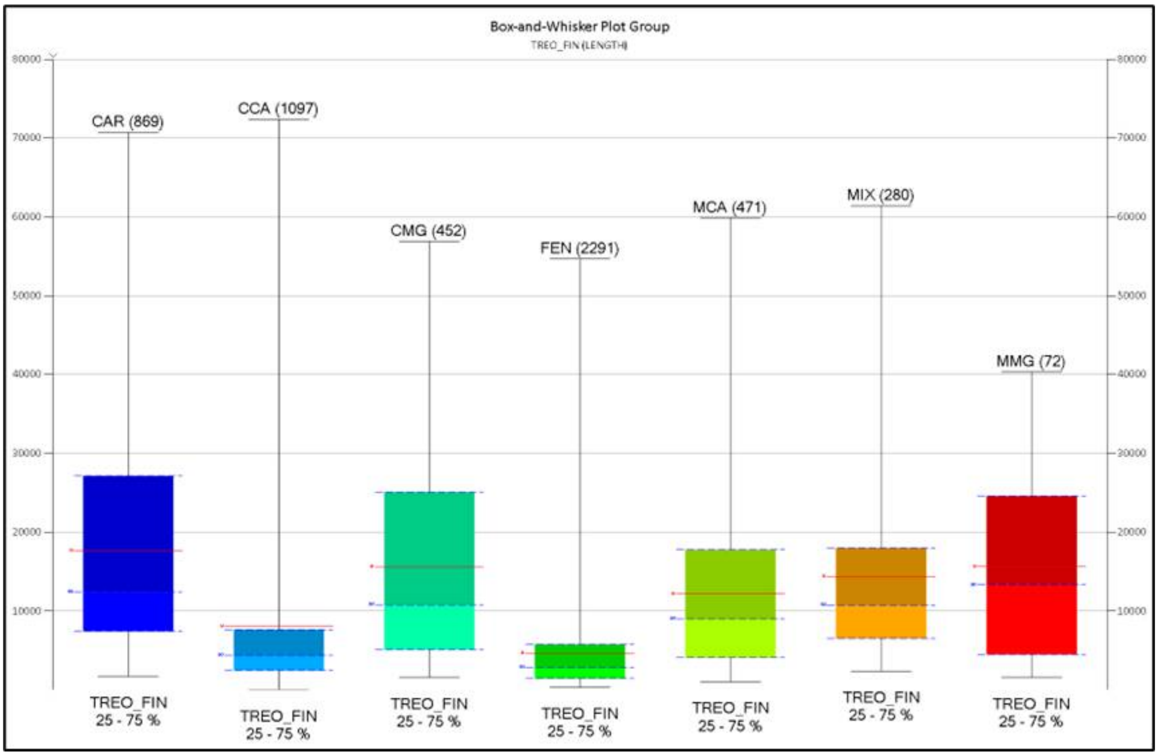
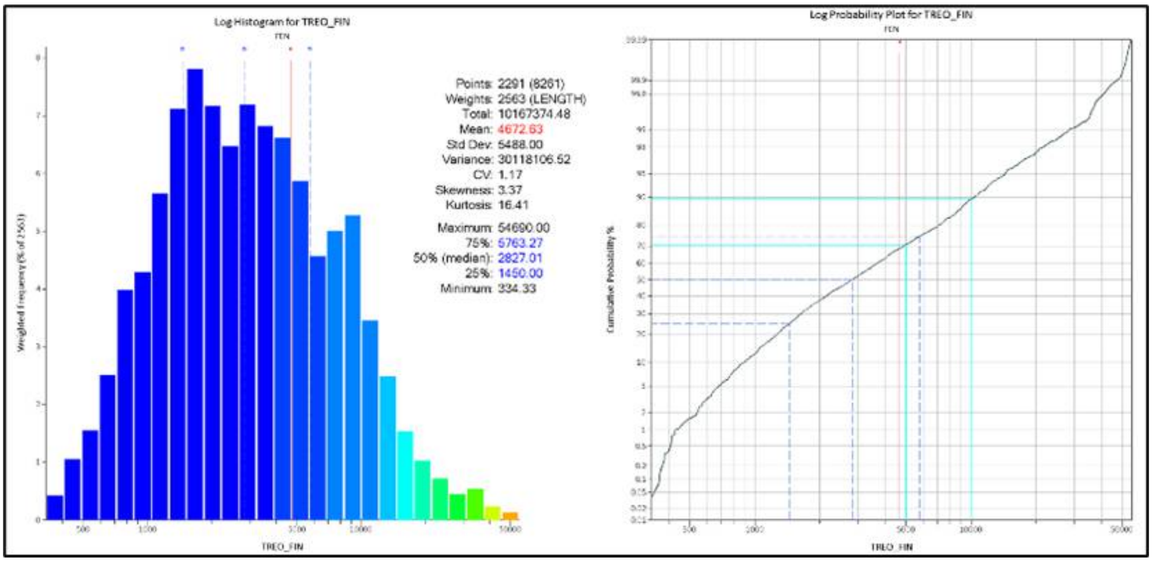


Figure 4.45 Global length-weighted TREO grades for material identified as fenite



Note: Log-histogram (left); log-probability plot showing 0.5 and 1.0% TREO cut-offs (right)

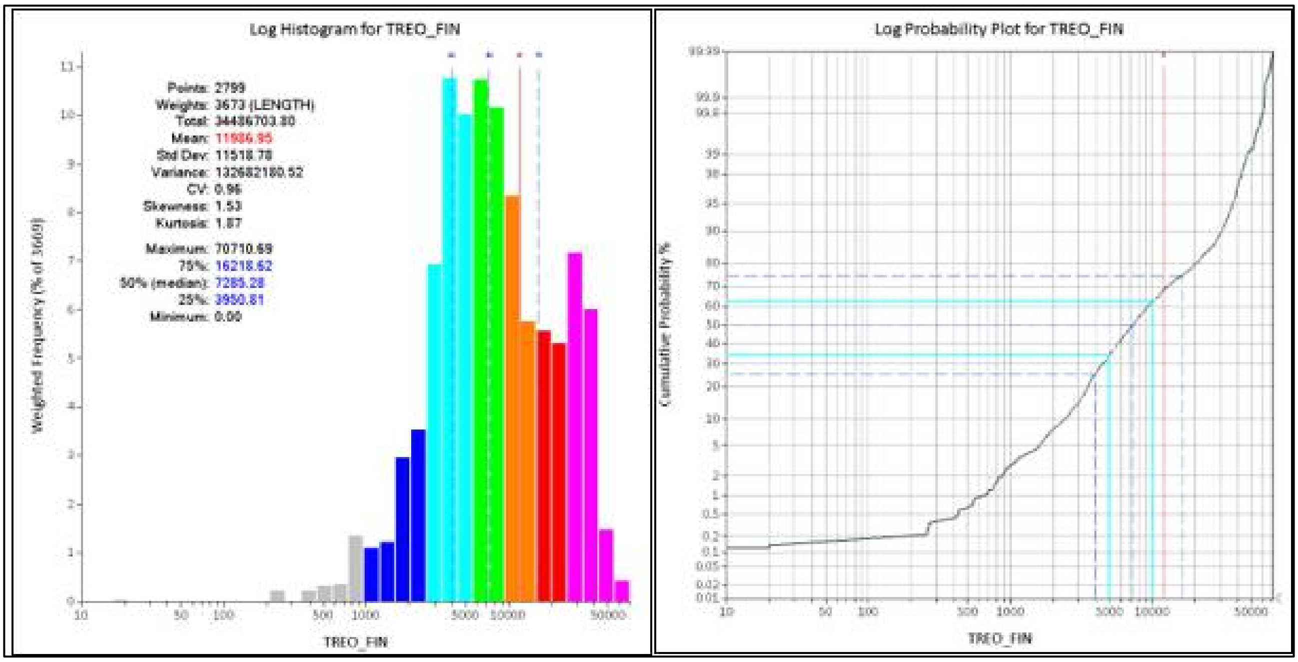
The length-weighted TREO grade distributions for Targets 1 and 4 were then reviewed. It was noted that there were multiple subtle grade inflections within the grade distribution, in reviewing all of the available information TREO grade cut-offs at 0.5% and 1.0% TREO were selected because:

- The inflection points of the grade distributions across Target 1 and 4 (Figure 4.46).

- Test work using TREO horizontal indicator variography for Target 1 assessing the impact of different TREO grade cut-offs. Below 0.5% TREO, the continuity fans all display similar patterns. However above 0.5% TREO, the grade continuity ranges start to reduce and there are subtle changes to the direction of maximum continuity. Above 1.0% TREO the larger scale continuity is significantly reduced, and the preferred continuity direction steepens.
- When reviewing the available data spatially, the rate of change in TREO grades incrementally increases across the 0.5% and 1.0% TREO grades.
- At these grade cut-offs, along strike and down dip continuity exists and these cut-offs overwhelmingly capture the mineralised samples.

Multiple mineralised shells at a variety of grade cut-offs were created initially using omni-directional interpolation parameters in Leapfrog. The 0.5% TREO cut-off provided the optimal cut-off to define the on-set of mineralisation, while the 1.0% cut-off defined spatially consistent high-grade domain along strike and down dip.

Figure 4.46 Target 1 TREO global grade distribution



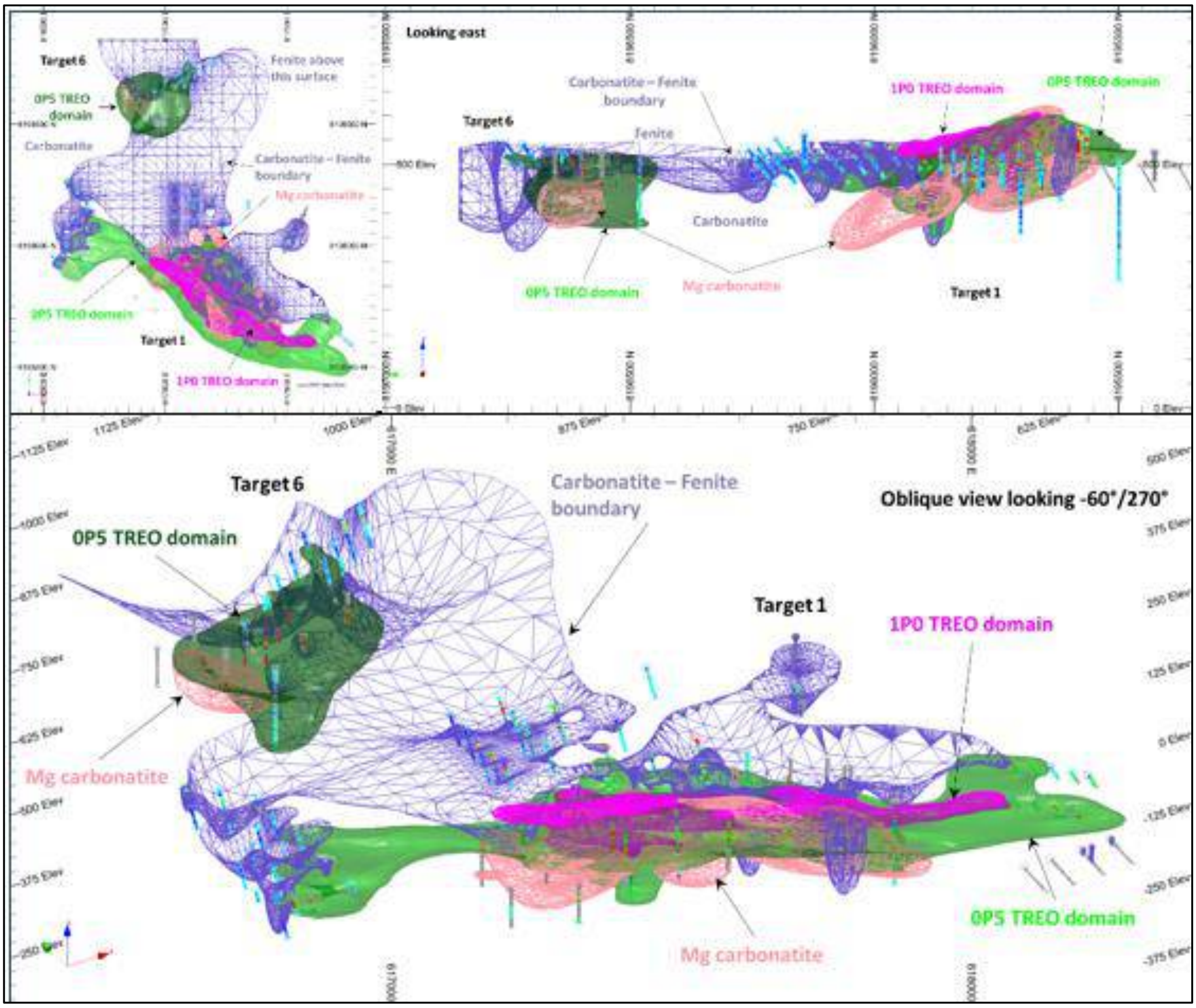
Note: Log-histogram (left); log-probability plot showing significant grade increments (right)

Geological modelling

All drill hole geological as well as the combined laboratory and handheld XRF assay data was used to model the geology and weathering, Geological and mineralisation modelling was undertaken using Leapfrog Geo (v2022.1) software. Initial geological interpretations were made to interpret a full rock model for both Target 1 and 4 (Figure 4.47). However, a combination of the drilling being clustered around the mineralisation, the lithologies being highly weathered, the available logging and the intrusive geological emplacement of the local geology combined to make producing consistent and robust geological interpretations extremely difficult. Hence, only generalised interpretations for the key rock types were undertaken being restricted to:

- A combined magnesium rich (>1% MgO) carbonatite (CMG) and mixed rocks (MMG).
- A carbonatite-fenite boundary surface.

Figure 4.47 Target 1 geological and main mineralisation interpretations



Note: Plan view (top left), section looking east (top right); oblique view looking down the mineralisation (bottom)

Final mineralisation interpretations were prepared at a 0.5% TREO and 1.0% TREO cut-off, using Leapfrog software, with a total of sixteen mineralised domains across all three target areas as listed in Table 4.16. This was completed by initially coding all samples with a ≥0.5% and ≥1.0% TREO indicators. If a sample was lower than the indicator cut-off it was flagged with a T*_0, and exceeded or equalled the indicator cut-off, it was flagged with a T*_1value (note the * refers to the respective target identity). This allowed the use of the more sophisticated geological modelling tools in Leapfrog Geo. Target 1 and 4 mineralised interpretations were created using the following workflow:

- Initially interpretations were created using generalised isotropic orientations.
- A variety of dip and dip direction anisotropies were then tested until relatively consistent 3-dimensional shapes was derived. The anisotropy was initially guided by the available indicator variography orientations, but then adjusted to provide a better local fit.
- The various shape editing tools in Leapfrog were used to refine the final and finalise triangulations to better reflect the data locally.
- Mineralisation at Target 6 was modelled using a simplistic horizontal trend because of the spatially limited data, with the available drill hole intersections being drilled at Target 6.
- In addition to the ESTDOM field, a DOMAIN field was included to facilitate the reporting of the 0.5 to 1.0% and ≥1.0% populations.

Table 4.16 Interpreted mineralised domains and estimation domains (ESTDOM and DOMAIN) coding

Target	Identifier	≥0.5% (DOMAIN = 0P5)		≥1.0% (DOMAIN = 1P0)	
		Wireframe	ESTDOM*	Wireframe	ESTDOM*
1	Main	wf_t1_0p5_main_2308	T1_01	wf_t1_1p0_mainclip_2308 wf_t1_1p0_hw_2308 wf_t1_1p0_fwclip_2308 wf_t1_1p0_nhgclip_2308	T1_11 T1_13 T1_14 T1_15
		wf_t1_0p5_nth2308	T1_02	N/A	
	Horizontal near surface	wf_T1_0P5_SG1_2308	T1_03	wf_t1_1p0_SG1aclip_2308 wf_t1_1P0_SG1bclip_2308	T1_16 T1_17
		wf_T1_0P5_SG2_2308	T1_04	wf_t1_0p5_sg2_2308	T1_18
4	T4 mineralisation	wf_t4_0p5_2308_1	T4_01	wf_t4_1p0_2308_1	T4_11
6	T6 mineralisation	wf_t6_0P5_2308	T6_01	wf_t6_1P0_1_2308 wf_t6_1P0_2CLIP_2308	T6_11 T6_12

Target 1

At Target 1, a total of four 0.5 to 1.0% TREO low grade and seven ≥1.0% TREO high grade mineralised domains were modelled (Figure 4.48). The main 0.5% TREO mineralised domain (T1_01) strikes towards 300-310° with dips ranging from sub-horizontal in the north-west, steepening to -35° in the south-east direction which coincides with the on-set of higher-grade mineralisation. The main low-grade domain has approximately 1,300 m strike length, with a maximum dip extent of 320 m and a maximum vertical extent of 220 m, narrowing down-dip. Within the main lower grade domain, there is development of four ≥1.0% TREO mineralised domains:

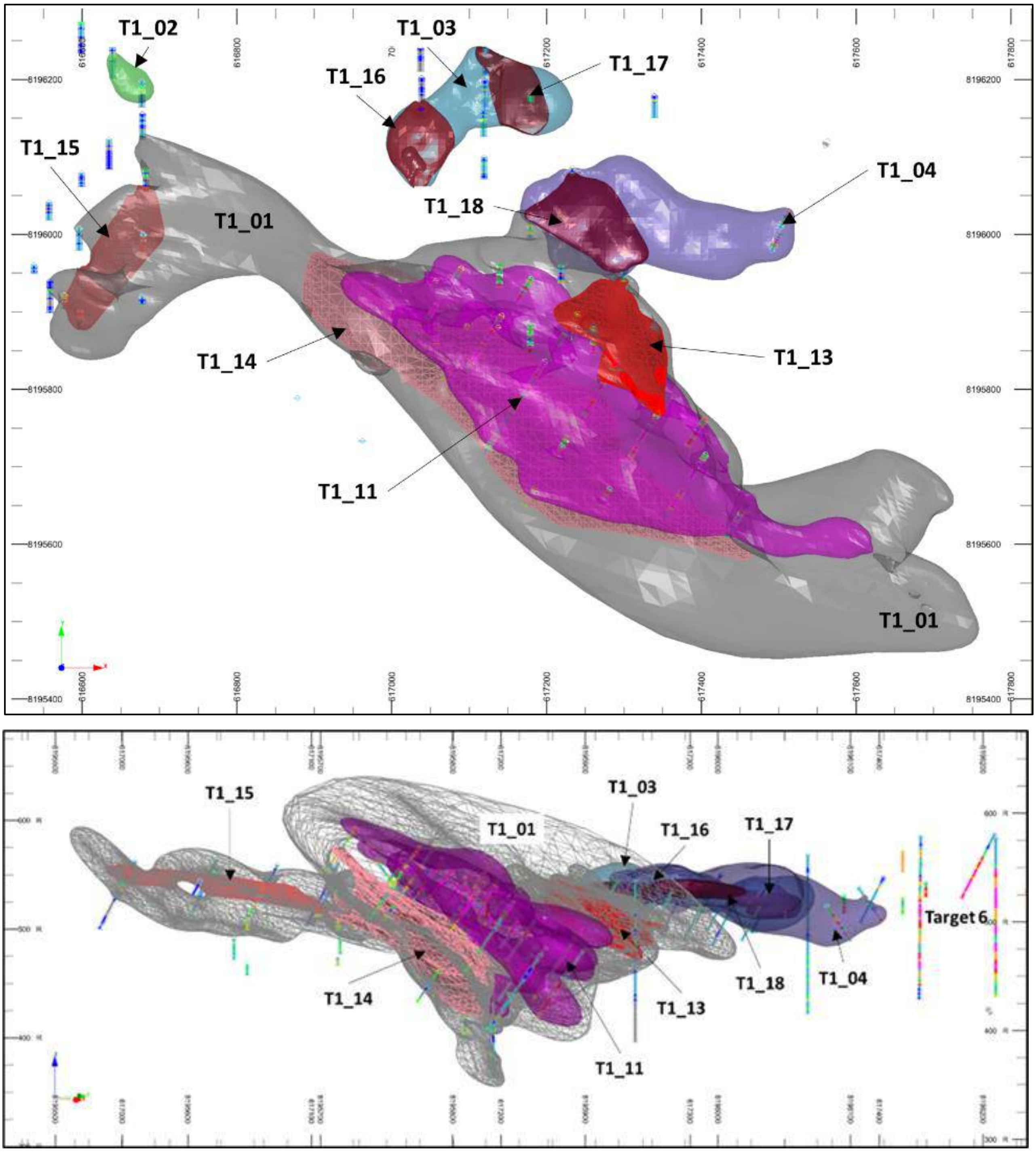
- The main high-grade domain (T1_11) presenting a higher-grade core to the enveloping low grade mineralisation, with a strike length of 775 m, pinching approximately 270 m down dip (220 m vertically) below surface.
- A small hangingwall mineralised position (T1_13) sub-parallel to the lower grade mineralised domain, with a strike length of 190 m, pinching approximately 115 m down dip, (100 m vertically) below surface.
- A narrow but consistent foot-wall mineralised position (T1_14) sub-parallel to the lower grade mineralised domain, with a strike length of 690 m, pinching approximately 200 m down dip (115 m) vertically.
- In the north-west margin of the low-grade mineralisation, there is development of a small flat lying ≥1.0% TREO mineralised domain sub-parallel to the local low-grade domain.

In addition to the northwestern extent of the main 0.5% TREO mineralisation domain, three additional sub-horizontal 0.5% TREO domains have been interpreted.:

- To the immediate north of the northwest margin of the main 0.5% grade domain, is a small mineralised low-grade domain (T1_02).
- Immediately north of the main 0.5% TREO domain is a moderate size low grade domain (T1_03), within which two small ≥1.0% TREO mineralised domains are interpreted (T1_16 and T1_17).
- To the northeast and adjacent to the main 0.5% TREO domain is a moderate size low grade domain (T1_04), within which a small ≥1.0% TREO mineralised domain (T1_18).

These flatter lying domains were interpreted to provide potential exploration targets and project scenario options going forward. The flatter lying orientations at Target 1 may reflect a spectrum of purely supergene or a combination of supergene and hypogene processes, but currently this is uncertain.

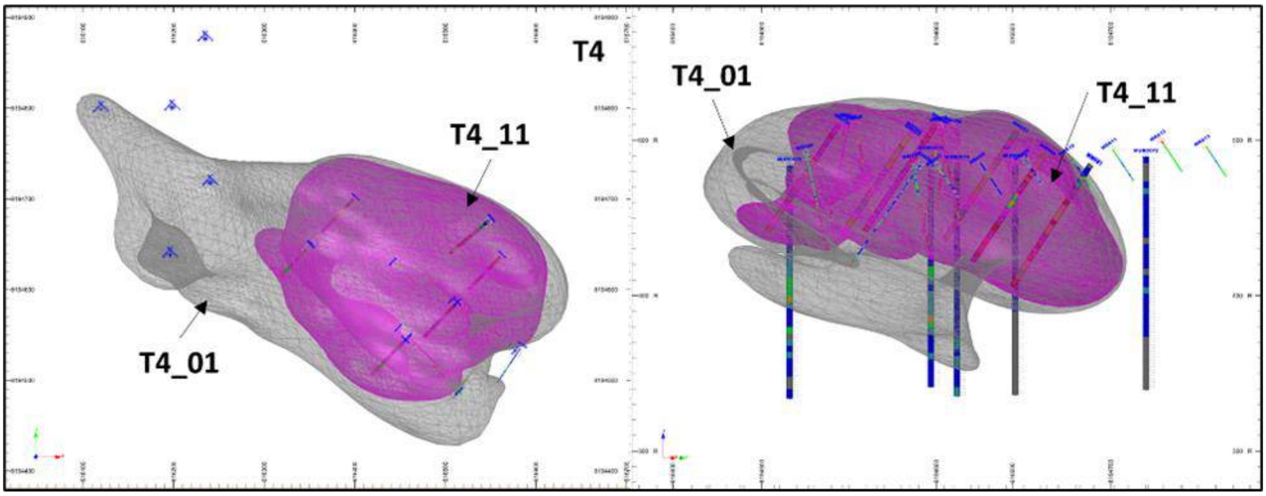
Figure 4.48 Target 1 interpreted mineralisation (top plan view, bottom section looking towards 305°)



Target 4

Target 4 consisted of a single broad 0.5 to 1.0% TREO low grade (T4_01), surrounding a single ≥1.0% TREO high grade mineralised domain (T4_11), presented in Figure 4.49. The Target 4 mineralisation boundaries are not well defined spatially, but within the boundaries the grades exhibit little variability. The mineralisation has been interpreted with a similar orientation as Target 1 (-25° towards 295°), primarily because it provided the best fit against the available data. The 0.5 to 1.0% TREO domain has a sub-horizontal lozenge shape, with a strike length of 585 m, and horizontal widths of 50 to 275 m across strike. It has a nominal down dip extent of 270 m, extending 110 m below surface, substantially thinning out and deepening towards the northwest. The ≥1.0% TREO mineralised domain has a strike length of 300 m and a down dip extent of 240 m, extending 230 m horizontally, and up to 95 m vertically from surface.

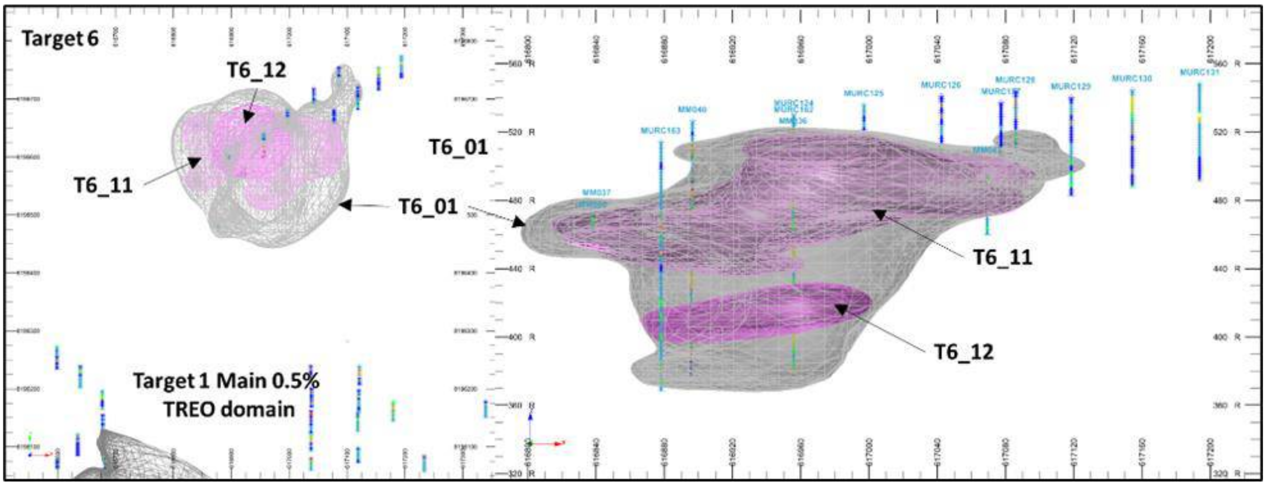
Figure 4.49 Target 4 interpreted mineralisation (left plan view, right section view looking towards 300°)



Target 6

Target 6 consists of a single 0.5 to 1.0% TREO low grade domain and two ≥1.0% TREO high grade mineralised domains (Figure 4.50). The ≥1.0% TREO are informed effectively by a single line of drilling which makes determining the mineralised geometry difficult with limited along strike information. The mineralisation is considered open in the northwest-southeast direction. Given the depth of the mineralisation at Target 6, it is likely to have similar controls as Target 1, however, this has not been confirmed. The 0.5 to 1.0% TREO low grade domain is approximately circular in shape, with horizontal length of 320 m in the northeast direction and 250 m towards the southeast. The vertical depth is approximately 380 m extending from approximately 10 m below surface. The ≥1.0% TREO domains are both similarly circular in shape and are separated by approximately 9 m of low TREO grade material.

Figure 4.50 Target 6 interpreted mineralisation (left plan view, right section view looking 000°)



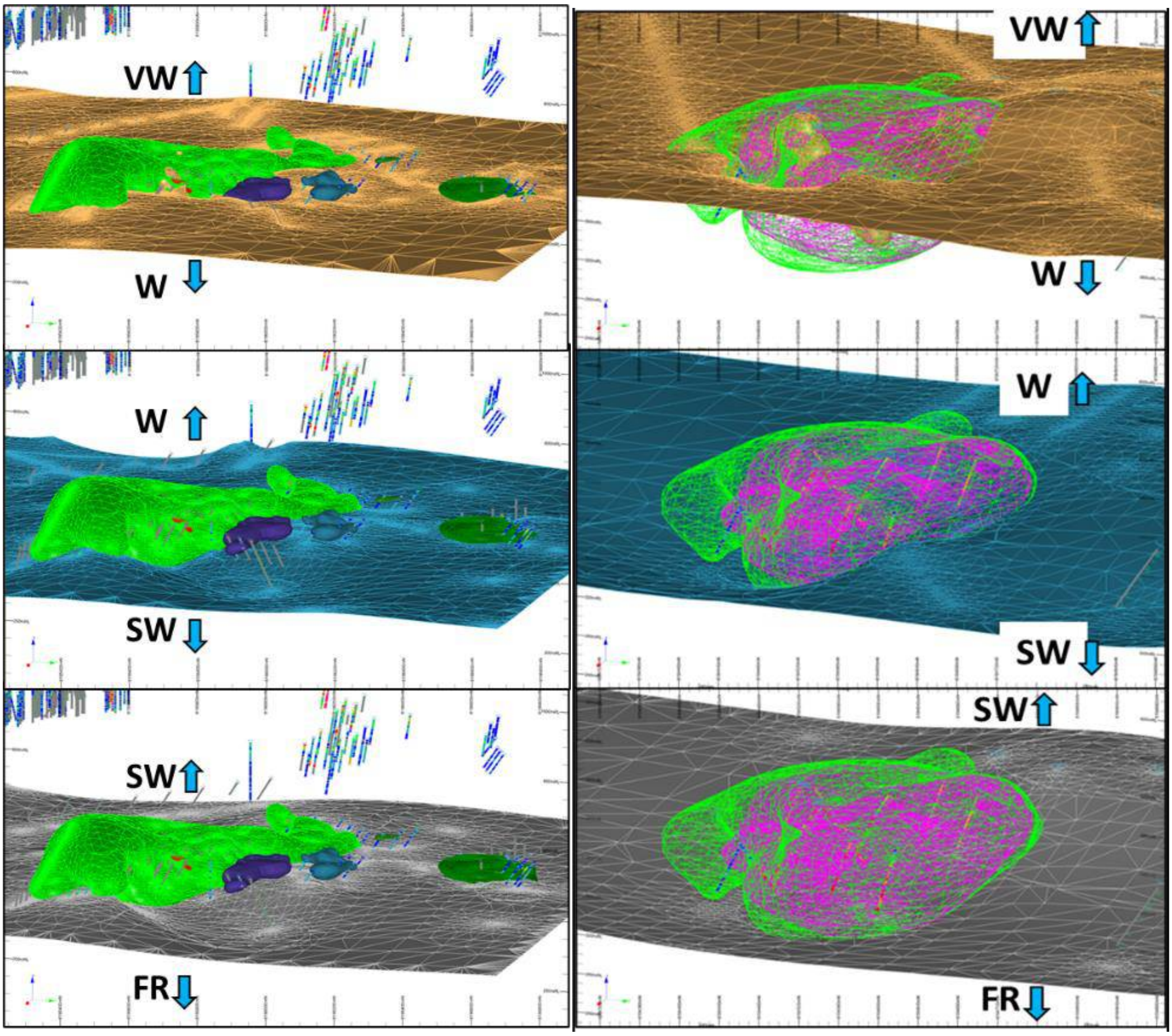
Weathering

Due to the need to define weathering for possible pit wall locations, the weathering was modelled well outside from mineralisation and encompassed most of the Project area. The Altona weathering code criteria was used to construct the interpretations (Table 4.17), using all available logging but ignored the Globe weathering data logged as either SP (saprolite) or TR (transitional). The logged weathering codes exhibited significant local variability and involved the grouping of different adjacent logged codes to generate interpretable shapes. However, the resultant shapes are broadly consistent in 3D (Figure 4.51).

Table 4.17 Interpreted weathering codes (WEATH)

Weathering codes	
WEATH Code	**Description**
FR	Fresh
SW	Slightly Weathered – weathered patches
W	Weathered
VW	Very Weathered – crumbling or loose
SO	Soil

Figure 4.51 Target 1 and 6 interpreted weathering surfaces (excluding SO) – oblique looking east



To test the veracity of the weathering interpretations, a series of box and whisker plots were generated for each of the mineralised and weathering domains. These confirmed that the weathering suitably captured different weathering geochemistry between the different weathering domains, Broader patterns identified included that transitioning from fresh to very weather, the TREO increased, while MgO and SO_3 decreased.

The depth of weathering combined with the magnesium depletion through the weathering profile makes modelling the carbonatite lithologies solely based on an MgO grade difficult. The apparent volume/ occurrence of MgO rich lithologies near surface will be reduced if solely using a MgO only grade criteria. This pattern is observed in the attempts to model the magnesium rich carbonatites where they appear to be domal structures that are not strictly expressed at surface. Hence it is almost certain that the interpreted MgO rich lithologies reflect an artefact of the weathering. Empirically the Monte Muambe REO mineralisation represents a relatively higher-grade central core (the ≥1.0% domains) which have been remobilised by weathering to create a lower grade halo around the higher-grade portions. Development of the sub-horizontal mineralisation at Target 1 and Target 6 may represent a dominantly supergene processes to create the patterns observed.

Data conditioning

Only laboratory assay data were used for the estimation at Monte Muambe. The sample data was flagged by respective ESTDOM and DOMAIN fields (Table 4.14) and the respective weathering codes (Table 4.15). Once validated all composite samples were created using the Datamine compdh composite function, controlled by the ESTDOM, DOMAIN and WEATH codes and using the parameters presented in Table 4.18. The resultant composites were then checked against the input statistics which suitably reflected the input statistics (Table 4.19).

Table 4.18 Composite creation parameters

Parameter	Value
Control fields	ESTDOM, WEATH, DOMAIN
Composite interval	2.0
Minimum length	0.5
Minimum gap	0.001
Maximum gap	0.0
Mode	Best fit (Mode = 1)

Table 4.19 Sample versus composite key statistics

Field	Target	Type	Domain	No. samples	Min.	Max.	Total	Mean	Standard deviation	Skewness	CoV
Length (m)	T1	Samples	0P5	583	0.15	4.00	1,229	2.11	0.82	0.96	0.39
			1P0	815	0.05	4.00	1,492	1.83	0.72	0.30	0.40
			WAST	588	0.27	4.00	1,649	2.80	1.16	-0.19	0.41
		Composite	0P5	624	0.93	2.65	1,229	1.97	0.12	-4.71	0.06
			1P0	754	1.00	2.11	1,492	1.98	0.08	-5.86	0.04
			WAST	837	0.85	2.35	1,649	1.97	0.12	-5.21	0.06
	T4	Samples	0P5	122	2.00	2.03	244.50	2.00	0.011	2.3	0.01
			1P0	182	1.50	2.00	362.00	1.99	0.061	-6.5	0.03
			WAST	525	1.80	2.00	1048.00	2.00	0.021	-7.8	0.01
		Composite	0P5	122	2.00	2.03	244.50	2.00	0.011	2.3	0.01
			1P0	182	1.50	2.00	362.00	1.99	0.061	-6.5	0.03
			WAST	525	1.80	2.00	1048.00	2.00	0.021	-7.8	0.01
	T6	Samples	0P5	147	0.08	4.00	216	1.47	1.30	1.01	0.88
			1P0	173	0.10	4.00	278	1.61	1.17	0.85	0.73
			WAST	233	0.04	4.00	600	2.57	1.60	-0.38	0.62
		Composite	0P5	108	1.00	2.18	216	2.00	0.11	-6.09	0.06
			1P0	140	1.75	2.05	278	1.99	0.06	-2.46	0.03
			WAST	301	1.00	2.43	599	1.99	0.08	-7.41	0.04
TREO %	T1	Samples	0P5	583	0.07	4.06		0.73	0.40	3.89	0.55
			1P0	815	0.00	7.07		2.19	1.18	0.68	0.54
			WAST	588	0.00	1.52		0.30	0.16	1.77	0.55
			0P5	624	0.08	4.06		0.73	0.39	3.97	0.53

Field	Target	Type	Domain	No. samples	Min.	Max.	Total	Mean	Standard deviation	Skewness	CoV
		Composite	1P0	754	0.19	6.47		2.20	1.16	0.71	0.53
			WAST	837	0.00	1.52		0.30	0.16	1.63	0.54
	T4	Samples	0P5	122	0.41	1.46		0.74	0.203	0.8	0.27
			1P0	182	0.28	5.68		2.13	1.221	0.8	0.57
			WAST	525	0.02	2.43		0.27	0.187	3.1	0.68
		Composite	0P5	122	0.41	1.46		0.74	0.203	0.8	0.27
			1P0	182	0.28	5.68		2.13	1.221	0.8	0.57
			WAST	525	0.02	2.43		0.27	0.187	3.1	0.68
	T6	Samples	0P5	147	0.13	5.37		0.75	0.56	4.32	0.75
			1P0	173	0.29	6.24		2.17	1.20	0.95	0.55
			WAST	233	0.08	2.88		0.33	0.18	1.63	0.54
		Composite	0P5	108	0.27	3.28		0.76	0.47	3.82	0.62
			1P0	140	0.53	5.68		2.16	1.10	0.97	0.51
			WAST	301	0.08	1.11		0.33	0.17	1.30	0.51

Note: CoV – Coefficient of variation

Statistics

Naïve statistics were reported by ESTDOM and are summarised in Table 4.20. Except for the Target 1 T1_11 population, samples exhibit low variability (coefficient of variation less than 1) and low coefficient of skew. The anomalous statistics for T1_11 domain in individual REEs (Not TREO) are the result of a single sample with very extremely high grades across most of the rare earth oxide variables.

Table 4.20 TREO composite statistics by ESTDOM

ESTDOM	No. samples	Min.	Max.	Mean	Standard deviation	Skew	CoV	Percentiles		
								25th	50th	75th
T1_01	532	820	36,544	7,104	3,189	3.2	0.45	5,382	6,638	8,457
T1_02	14	5,709	9,978	7,488	1,507	0.3	0.20	5,734	7,261	8,585
T1_03	42	1,822	14,465	5,917	2,794	1.1	0.47	3,732	5,896	6,884
T1_04	36	2,193	40,638	11,549	9,075	2.0	0.79	6,412	7,610	11,704
T1_11	584	1,895	61,393	23,174	11,439	0.4	0.49	13,381	22,278	31,361
T1_13	40	9,010	64,664	29,418	14,229	0.6	0.48	17,512	28,371	36,444
T1_14	68	6,792	51,810	14,934	7,459	2.4	0.50	10,239	12,828	17,054
T1_15	20	9,869	27,121	15,560	5,874	1.0	0.38	11,162	12,917	17,622
T1_16	20	5,174	23,574	13,468	5,475	0.6	0.41	10,300	11,828	16,959
T1_17	14	4,909	14,110	11,292	2,984	-1.1	0.26	10,408	11,020	13,813
T1_18	8	10,253	14,969	12,453	1,683	0.3	0.14	10,253	12,295	12,536
T1_W	837	0	15,244	3,019	1,624	1.6	0.54	1,890	3,001	3,803
T4_01	122	4,123	14,577	7,384	2,030	0.8	0.27	5,514	7,222	8,688
T4_11	182	2,834	56,756	21,270	12,211	0.8	0.57	10,842	17,662	29,575
T4_W	525	216	24,279	2,747	1,867	3.1	0.68	1,074	2,919	3,802
T6_01	108	2,681	32,760	7,560	4,698	3.8	0.62	5,253	6,409	8,449
T6_11	115	8,431	56,824	21,026	10,642	1.2	0.51	12,405	17,670	26,895
T6_12	25	5,271	47,335	24,397	12,372	0.3	0.51	12,642	19,283	34,501
T6_W	301	827	11,080	3,261	1,677	1.3	0.51	1,950	3,141	4,085

Note: CoV – Coefficient of variation

The grade distribution plots for the T1_01, T1_11, T4_01 and T4_11 mineralised domains are presented Appendix D. A total of thirteen composites had top-cuts applied, to reduce the impact of extreme grades for a very limited number of composites as presented in Table 4.21.

Table 4.21 Applied top-cuts

ESTDOM	Variable	Samples	Uncut samples				Top-cut samples				
			Max.	Mean	Std. devn.	CoV	Value	No. of cut	Mean	Std. devn.	CV
T1_11	CE_2O_3	584	539,234	11,474	22,510	1.96	**32,000**	1	10,606	5,444	0.51
	DY_2O_3	584	19,573	112	807	7.22	**2,500**	1	83	105	1.27
	TB_2O_3	584	390	16.0	16.8	1.05	**150**	1	15.5	8.4	0.54
T6_01	TREO	108	32,760	7,560	4,698	0.62	**20,000**	3	7,230	3,134	0.43
	CE_2O_3	108	16,529	3,152	2,432	0.77	**9,000**	3	2,959	1,511	0.51
	PR_2O_3	108	1,512	346	210	0.61	**1,100**	3	338	168	0.50
T6_12	NB_2O_5	25	1,466	306	300	0.98	**750**	1	278	208	0.75

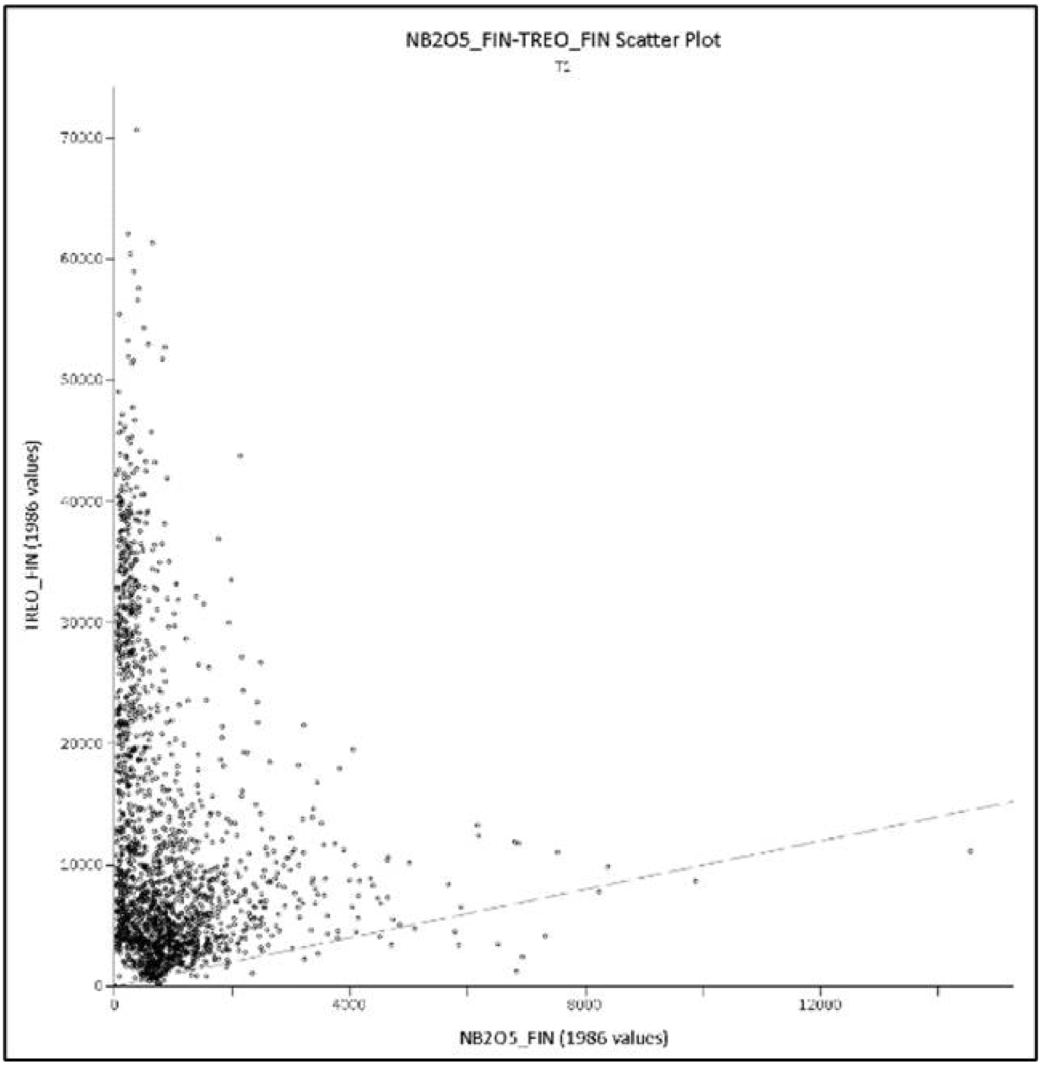
Note: CoV – Coefficient of variation; Std devn. – Standard deviation

Pearson correlation coefficients were derived for the key variables at Monte Muambe. For all target-domain combinations the correlations are similar. However, there some key differences:

- At Target 1, Ce_2O_3 behaves differently in the two domains. For TREO there is good correlation in the lower grade domain, but a poor correlation with TREO in the higher-grade domain. The correlation with Ce_2O_3 are different between the two domains.
- Target 4 does not exhibit the same correlation patterns as Target 1. Ce_2O_3 is strongly correlated with TREO in both the lower and higher-grade domains, while Nb_2O_5 exhibits the most significant difference compared to Target 1. Within the lower grade domain there are no significant Nb_2O_5 correlations, however, in the higher-grade domain, Nb_2O_5 exhibit weak to moderate correlations with TREO and Pr_2O_3. Both domains exhibit moderate to good correlations between Tb_2O_3 and Th.
- Target 6 exhibits similar correlation patterns between the lower and higher-grade domains, with the exception of the behaviours of Dy_2O_3 and Nb_2O_5, Dy_2O_3 and U as well as Nb_2O_5 and U. Target 6 exhibits similar correlations as Target 4 and does not reflect the discrepancy observed at Target 1 between the lower and higher grade domains for Ce_2O_3. Target 6 and Target 4 exhibit similar correlation patterns between Tb_2O_3 and Th which are absent from Target 1. At Target 6, Nb_2O_5 exhibits better correlation with U than observed at Target 4.

Further to these observations, a scatterplot was prepared for Nb_2O_5 and TREO (Figure 4.52) that highlights the two sub-populations within the mineralisation; a low Nb_2O_5-high TREO sub-population, and a high Nb_2O_5-low TREO sub-population.

Figure 4.52 Nb_2O_5 – TREO scatter plot for target 1 (combined mineralised domains)



These correlation observations highlight differences between the respective targets, which is likely to represent differences in the primary genesis of the mineralisation, and subsequent impact of weathering at Monte Muambe.

Variography

Variography was completed using Snowden Supervisor v8.15.0.3 software. Variogram modelling was prepared for the two major 0.5 - 1.0% and ≥1.0% domains at target 1, and the two estimation domains at target 4 (T1_01, T1_11, T4_01 and T4_11 respectively) as these were the key domains, as well as being the only domains with sufficient samples. The variography for all variables were modelled separately, although efforts were made to align the variogram directions.

For TREO at Target 1, the 0.5 - 1.0% estimation domain horizontal plane had three potential directions that could be selected: 010°, 050° and 320°. Although the 320° direction approximated the interpreted mineralised domain but had a shorter range (130 m compared to 190 m) than the other two directions, and for the 320° the subsequent across-strike and dip plane variography were very poorly structured. The 010° direction was selected as it provided variograms which were better structured in the across strike and dip plane directions and with longer ranges. The nugget structure for the back-transformed variograms were variable, with TREO and Ce_2O_3 having elevated nuggets greater than 60%, Tb_2O_3 had a nugget of 43%, while the rest all had low nuggets less than 15%.

The ≥1.0% estimation domain had single preferred orientation along 105°/285°, broadly sub-parallel but somewhat oblique to the interpreted ≥1.0% estimation domain. The Target 1 variogram models were moderately to well structured, with nugget structures less than 37% for all variables except Ce_2O_3 and Nd_2O_3 with nuggets of 66% and 47% respectively.

For Target 4, the horizontal variogram fans were poorly structured, such that the selected variogram directions were primarily based on the geological interpretations and the modified to best fit the respective variogram fans. The final selected orientations were broadly similar for each domain, with a relatively flat dip planes.

The nugget structures for the 0.5-1.0% are mixed, with TREO, Ce_2O_3, Pr_2O_3 and Nb_2O_5 having elevated nuggets of between 26 to 54% of the sill, with the rest having low nuggets less than 20%. For the ≥1.0% estimation domain the nugget structures were uniformly low (less than 20%) with the exception of Nd_2O_3.

The different variogram patterns between the 0.5-1.0% and the ≥1.0% estimation domain indicate the presence of different controls of mineralisation at Monte Muambe between the two estimation domains.

No variography was undertaken for the Target 6 mineralisation because of the limited number of samples.

The modelled variograms are contained in Appendix E.

Block model and resource estimation

A single block model prototype was used for both Target 1 and Target 4 block models, allowing the two models to be easily combined if required. The parent block size was derived from kriging neighbourhood analysis (KNA) for the target 1 TREO ≥1.0% estimation domain.

Equal easting and northing parent block sizes was preferred with the selection of a 20 mE x 20 mN x 5 mRL block size. This is a compromise between obtaining good estimation performance, suitable block filling of the interpretation and suitable resolution for mine design/mine planning purposes.

The final selected block configuration is presented in Table 4.33. The comparison between the wireframe and block model volumes (Table 4.22) demonstrates that the block model has suitably captured the interpreted volumes for both estimation domains.

Table 4.22 Monte Muambe Target 1 and Target 4 block model configuration

Item	Easting (mX)	Northing (mY)	Elevation (mRL)
Origin	615,900	8,194,200	335
Extent	617,800	8,197,000	665
Parent block size	20	20	5
Nos parent blocks	95	140	66
Minimum sub-cell	2.0	2.0	0.5

Table 4.23 Wireframe – block model fill comparison (not clipped to topography)

ESTDOM	Wireframe volume (m^3)	Block model (m^3)			Percent difference (%)
		0.5 – 1.0% TREO	≥1.0% TREO	Total	
T1_01	22,614,617	17,643,788	4,957,713	22,601,501	-0.06%
T1_02	21,778	21,798		21,778	0.00%
T1_03	548,407	394,730	154,004	548,734	0.06%
T1_04	1,235,054	1,118,674	117,015	1,235,689	0.05%
T4_01	7,487,217	4,226,631	3,260,772	7,487,403	0.00%
T6_01	4,625,022	3,398,174	1,227,588	4,625,762	0.02%
0.5 – 1.0% TREO	**36,532,095**	**26,803,795**	**9,717,092**	**36,520,888**	**-0.03%**
T1_11*	4,170,432			4,170,350	0.00%
T1_12*	15,010			14,984	-0.18%
T1_13*	303,048			302,990	-0.02%
T1_14*	347,293			347,266	-0.01%
T1_15*	136,939			136,822	-0.09%
T1_16	86,291			86,288	0.00%
T1_17	67,713			67,660	-0.08%
T1_18	117,015			117,020	0.00%

ESTDOM	Wireframe volume (m³)	Block model (m³) 0.5 – 1.0% TREO	≥1.0% TREO	Total	Percent difference (%)
T4_11	3,260,875			3,260,772	0.00%
T6_11	968,209			968,150	-0.01%
T6_12	259,379			259,416	0.01%
≥1.0% TREO	**9,732,206**	**-**	**-**	**9,731,718**	**-0.01%**
Total	**46,264,301**	**26,803,795**	**9,717,092**	**46,252,605**	**-0.03%**

Contact analysis

Contact analysis was undertaken to test the boundary conditions for estimation, which demonstrates that for estimation purposes, the boundaries should be treated as hard boundaries. Contact analysis was also performed by estimation domain for the weathering boundaries, with all boundaries behaving as soft or no boundaries.

Grade Interpolation and estimation parameters

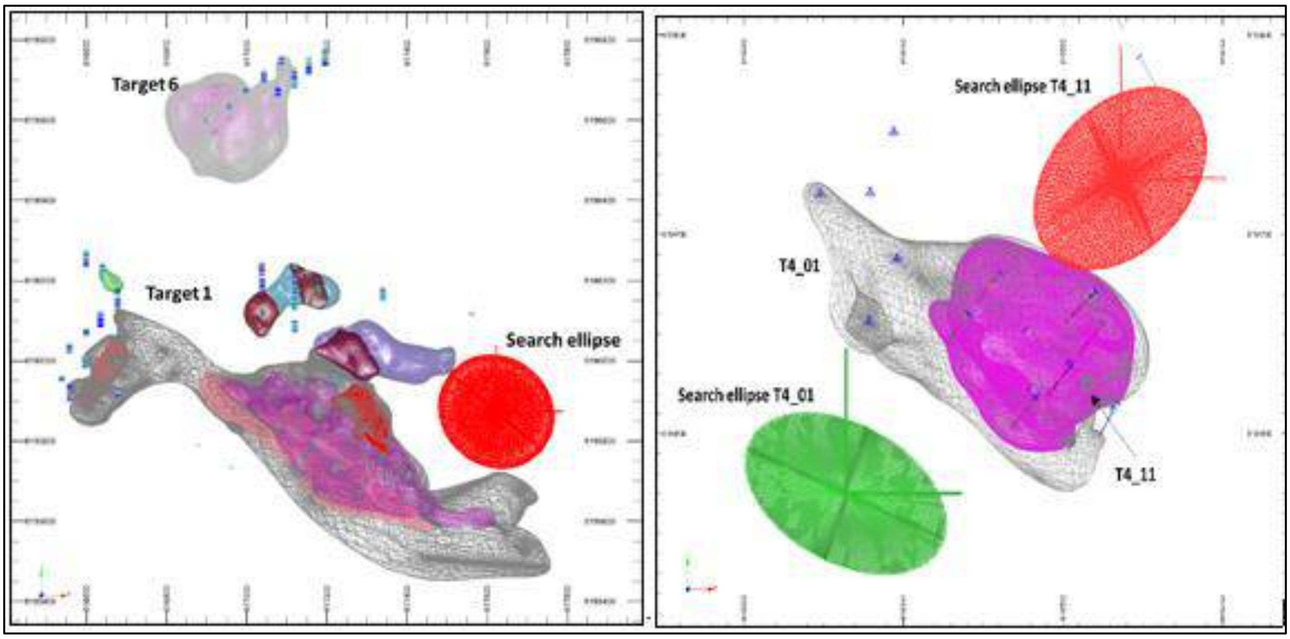
As a function of the low-grade variability, ordinary kriging of the 2.0 m composites was selected as the most appropriate grade estimation technique, using the top-cut values where applied. The 0.5-1.0% and ≥1.0% TREO boundaries were treated as hard boundaries, with estimation into parent blocks.

All variables within a mineralised domain used identical search parameters to ensure the cross-correlation between variables were maintained. The final search directions were based on a combination of:

- The final search orientation approximated the average interpreted geometry in combination with the average modelled variogram directions.
- The first and second search ranges were kept identical in the plane of the mineralisation to manage the different variogram orientations of individual variables.

The search ellipses used for estimation are presented in Figure 4.53.

Figure 4.53 Search ellipse Target 1 and 6 (left) and Target 4 (right)



Three search passes were used for estimation to ensure all blocks received an estimate.

For T1_01 and T1_11 domains, a maximum of four samples per drill hole was applied to ensure more than one drill hole informed the estimate. The other target/domain combinations did not have sufficient drilling to warrant using these criteria.

The search parameters are summarised in Table 4.24.

Initial review of the block model estimation passes:

- The estimation domain pass volume comparison confirms the majority of the mineralisation was informed by search pass 1 (Table 4.26).
- 84% of the block model by volume was informed in the first pass, an additional 13% informed in the second pass, and 2% being informed by the third pass. This also confirmed that small proportion the T1_01 and T1_04 estimation domains did not receive an estimate (Figure 4.54). This was a function of:
  - The interpretation of the southeastern lobe of T1_01 was informed by portable XRF assay data, which was removed from the estimation data set, resulting in no estimate
  - At T1_04, the available drilling is too sparse to inform this area of the domain.

Unestimated cells were assigned default grades lowest value from the most appropriate search pass average, and the respective search pass re-set to 99 (Table 4.25).

Figure 4.54 Target 1 model coloured by search pass

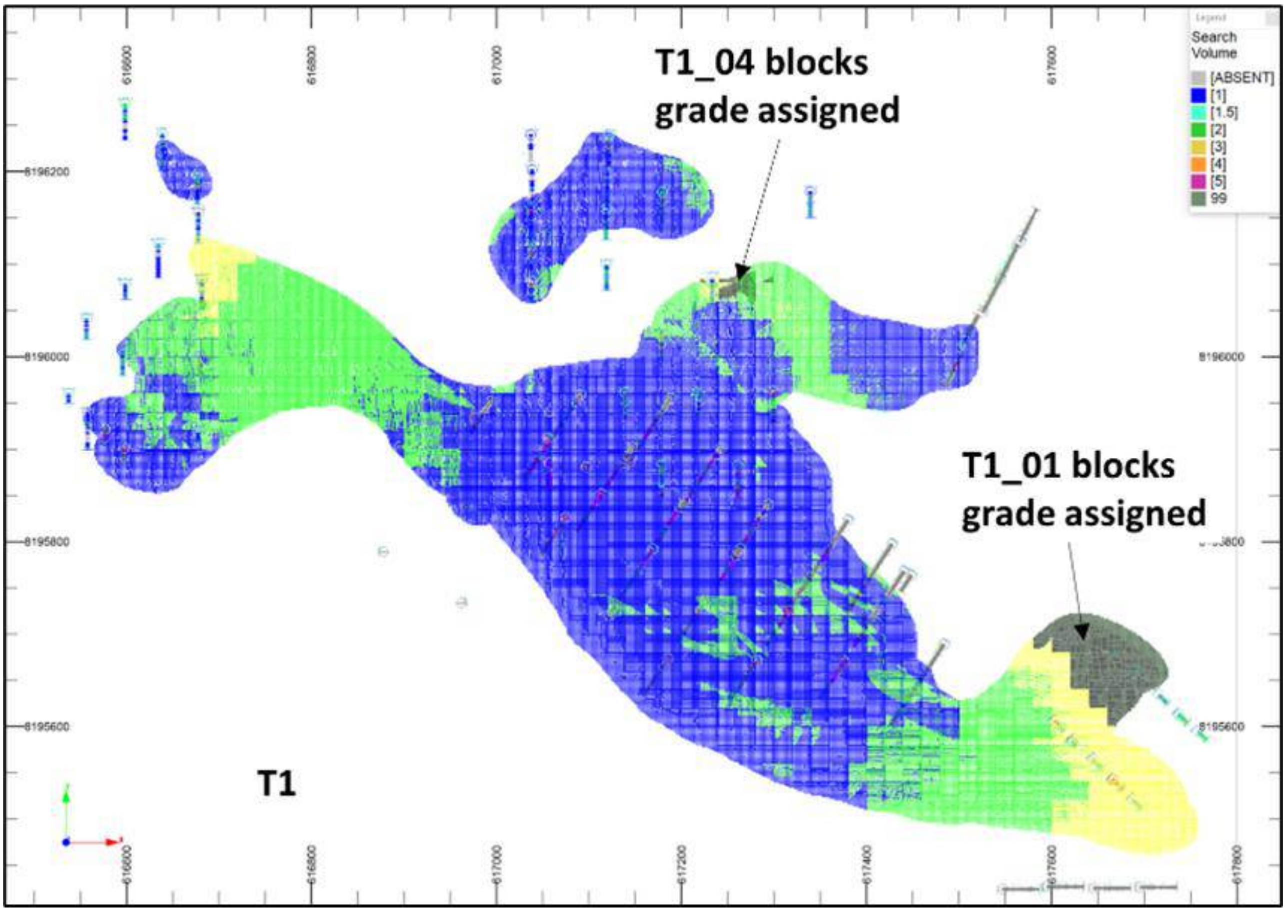


Table 4.24 Search parameters for estimation

Target	Domain	ESTDOM	Search reference	Search type	Datamine		Search pass 1		Search pass 2		Search pass 3		Samples / drill hole
					Rotation	Axis	Distance	No. of samples	Distance	No. of samples	Distance	No. of samples	
1	0.5 - 1.0%	T1_01	1	2	-140 -25 0	3 1 3	150 150 15	8-24	300 300 30	8 – 24	450 450 45	4-12	4
1	≥1.0%	T1_11, T1_13, T1_14	11	2	-140 -25 0	3 1 3	150 150 15	8 – 24	300 300 30	8 - 24	450 450 45	4-12	4
1 and 6	Minor domains	T1_02, T1_03, T1_04, T1_15, T1_16, T1_17, T1_18, T6_01, T6_11, T6_12	20	2	-140 -25 0	3 1 3	150 150 15	4-20	300 300 24	4-20	300 300 24	2-12	N/A
4	0.5 - 1.0%	T4_01	1	2	0 0 110	3 1 3	90 165 60	8-20	180 230 120	8-20	270 495 180	8-12	NA
4	≥1.0%	T4_11	11	2	-60 160 -30	3 1 3	150 100 80	8-20	300 200 160	8-20	450 300 240	8-12	N/A

Table 4.25 Grades assigned to blocks not estimated after three search passes

ESTDOM	TREO ppm	Ce_2O_3 ppm	Dy_2O_3 ppm	Nd_2O_3 ppm	Pr_2O_3 ppm	Tb_2O_3 ppm	Nb_2O_5 ppm	Th ppm	U ppm
T1_01	6,700	2,780	75	955	285	15	870	195	20
T1_04	5,700	2,300	60	675	220	10	760	225	9

Table 4.26 m_ m_t1_t6_2309and m_ m_t4_2309 estimation pass volume comparison

ESTDOM	Pass 1		Pass 2		Pass 3		Pass 4		Total volume (m^3)
	Volume (m^3)	%	Volume (m^3)	%	Volume (m^3)	%	Volume (m^3)	%	
T1_01	7,120,348	71.6%	2,137,144	21.5%	550,002	5.5%	131,884	1.3%	9,939,378
T1_02	21,790	100.0%	8	0.04%	-		-		21,798
T1_03	280,630	94.8%	15,332	5.2%	-		-		295,962
T1_04	346,056	46.6%	345,756	46.5%	48,114	6.5%	3,036	0.4%	742,962
T1_11	3,723,614	98.0%	71,408	1.9%	5,572	0.1%	-		3,800,594
T1_13	302,566	99.9%	424	0.1%	-		-		302,990
T1_14	214,532	61.8%	130,690	37.6%	2,044	0.6%	-		347,266
T1_15	98,902	77.5%	28,700	22.5%	-		-		127,602
T1_16	78,278	100.0%	22	0.03%	-		-		78,300
T1_17	66,924	98.9%	736	1.1%	-		-		67,660
T1_18	70,380	77.8%	20,070	22.2%	12	0.0%	-		90,462
T1 total	**12,324,020**	**77.9%**	**2,750,290**	**17.4%**	**605,744**	**3.8%**	**134,920**	**0.9%**	**15,814,974**
T4_01	3,508,544	93.1%	261,658	6.9%	-		-		3,770,202
T4_11	3,113,842	100.0%	-		-		-		3,113,842
T4 total	**6,622,386**	**96.2%**	**261,658**	**3.8%**	**-**		**-**		**6,884,044**
T6_01	2,840,572	83.6%	557,380	16.4%	-		-		3,397,952
T6_11	968,150	100.0%	-		-		-		968,150
T6_12	257,710	99.3%	1,706	0.7%	-		-		259,416
T6 total	**4,066,432**	**87.9%**	**559,086**	**12.1%**	**-**		**-**		**4,625,518**
Total	**23,012,838**	**84.2%**	**3,571,034**	**13.1%**	**605,744**	**2.2%**	**134,920**	**0.5%**	**27,324,536**

Block model validation

Initial validation of the estimate was by visual comparison between the input composites and the estimate in section and plan view (Figure 4.55 and Figure 4.56), and there is a good correlation between the composite and estimated grades spatially.

Figure 4.55 Target 1 visual validation in section and plan view

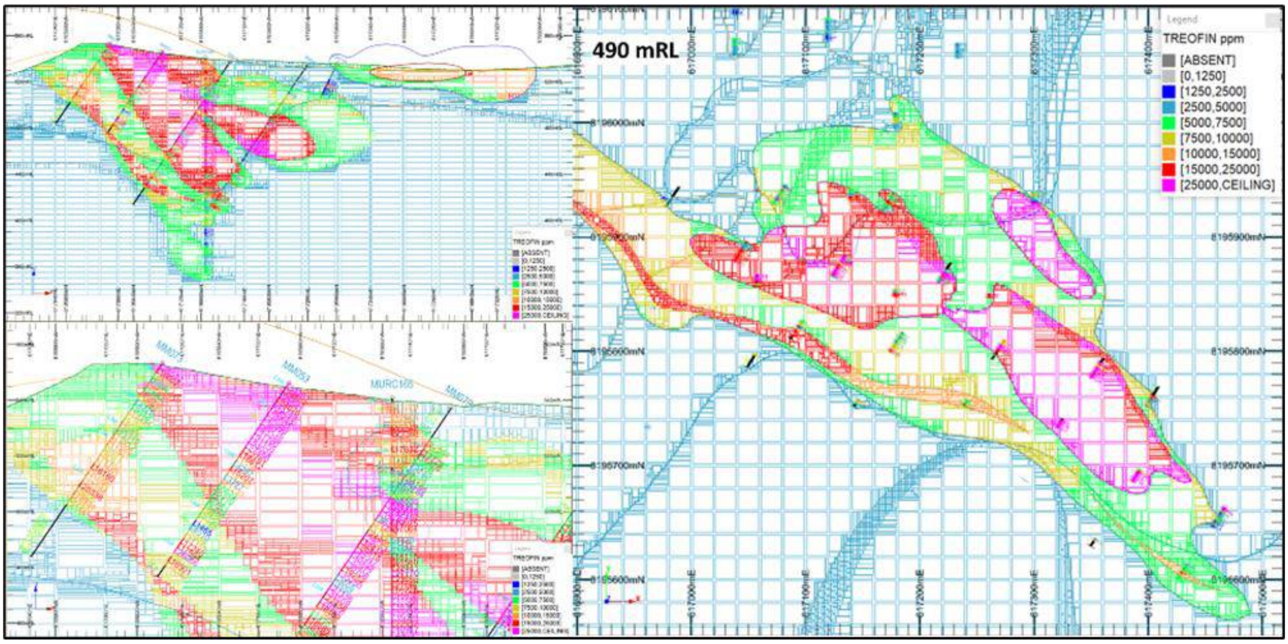
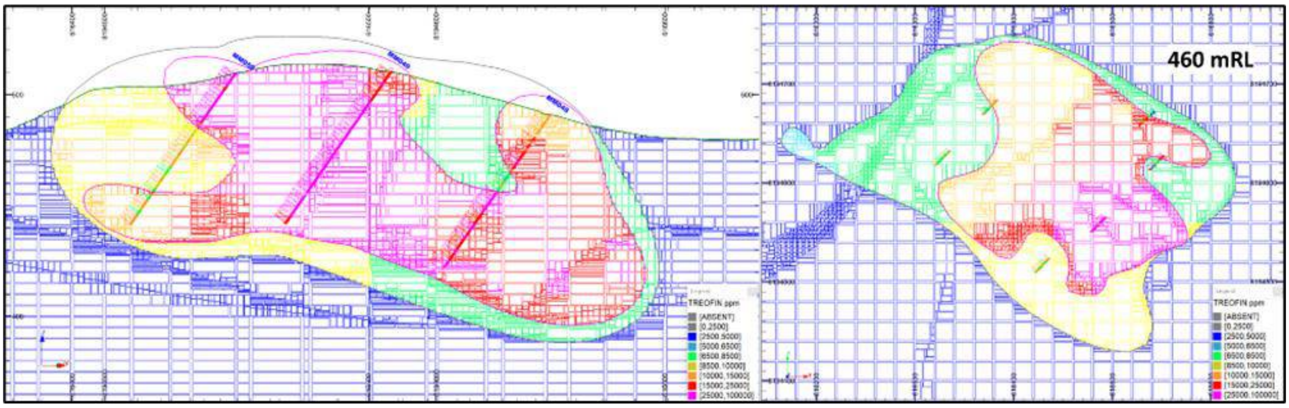


Figure 4.56 Target 4 visual validation in section and plan view



The whole of domain average naïve and declustered grades were then compared with the block model average grades and there was also good correlation between the composites and estimated grades (Appendix F).

Trend plots were then prepared for the naïve and declustered composite grades and the block model averages. An inverse distance cubed (ID^3) test estimate was also prepared for validation purposes using identical estimation parameters as the ordinary kriged estimate. There is good correlation between the composite samples, the ID3 test estimate and the ordinary kriged estimate (Appendix F).

4.8.3 Density

The diamond drilling programme in 2021 (five boreholes) had as one of the prime objectives the recovery of core for in situ density measurement. Due to the variable porosity of the carbonatite and weathered nature of much of the rock intersected several different density measurement methods were investigated.

Rock intersected in the diamond boreholes was interpreted as comprising 4 types from a density aspect:

- Type 1 - samples with low to high porosity and no cavities.
- Type 2 - samples with low to high porosity, with cavities outcropping at the surface of the cores.
- Type 3 - weathered cores with cracks, largely made of clay and/or limonite, still holding, but that would crumble in water or during cutting using a core cutter.
- Type 4 - crumbling cores with no integrity or competency.

The following density measurement methods were considered and, in some cases, tested:

- The gas pycnometer method was discarded due to the high porosity of many samples.
- The cling film immersion method was tried and discarded as a lot of air was getting trapped between the sample and the film.
- The wax immersion method was discarded due to its complexity and the equipment involved.
- The saturated immersion method was tried and was found to work well for Type 1 samples, but posed difficulties for samples having cavities outcropping at the surface of the drill holes.
- The calliper method was found to be suitable for Type 1 and Type 2 samples and was used as the main density measurement method. Cylinders were cut using the core saw. Several measurements of the diameter and the length of each cylinder were taken and averaged, and the cylinders were weighed.
- For Type 3 samples, a variation of the calliper method was used. A cylinder as perfect as possible was cut using a knife, and (due to some swelling of the core), the density calculation used the nominal inner diameter of the bit as opposed to the measured diameter of the core.
- For Type 4 samples, no density measurements were possible. The core tray method was considered but it was discarded because of the difficulty of ascertaining the actual core length in the tray.

Weights were measured using a newly purchased density measurement scale with an accuracy of 0.1 g. No QC system was in place when the measurements were made.

The density measurement procedure is covered by SOP 2021-03. In total, 371 density readings were taken from 590 m of core prior to cutting with a diamond saw. Each core piece used for density measurement was marked and replaced in the core tray. All density measurements were carried out on air dried core. In August 2023, at the request of the visiting Snowden-Optiro CP representative, a number of checks were carried out to verify the 2021 measurements and compensate for the lack of QAQC at the time of the original density measurements in 2021.

A set of reference weights was purchased and used to check the scale used for the density measurements. Certificates have been requested from the supplier and will be forwarded to the CP when received. Results of the checks are summarized in Table 4.27 below:

Table 4.27 Check weight scale results

Check weight (g)	Weight measured by scale (g)
200	200.0
500	500.1
1,000	1,000.2
2,000	2,000.5

While a drift was noted (and will be corrected in future uses of the scale), it is not significant in relation to the core sample weights concerned. For a 2 kg sample with a density of 2.5 $g.cm^3$, a 0.5 g error on the weight would correspond to a 0.025% error on the density.

The density of 20 samples was rechecked. It must be noted that the samples had been split and were now half-cores, and that some chipping on the edges of some samples had occurred.

Two density measurements were used:

- The calliper method, with several measurements of the length, diameter, and thickness of each sample.
- The saturated immersion method.

Half-core calliper method summary

- For each half-core:
  - The diameter (d) of the half-core was measured using a calliper in 3 different points, and its mean was calculated.
  - The thickness (t) of the half-core was measured using a calliper in 3 or 4 different points, and its mean was calculated.
  - The length (l) of the core of the half-core was measured with a tape measure along three different lengths, and its mean was calculated.
  - The weight (w) was measured on a scale with a 0.1 g accuracy.

Where thickness was > radius, the following formula was used to calculate the density:

$$\frac{w}{((\Pi \times (d/2)2/2) + (d \times (t-d))) \times l}$$

Where thickness was < radius, the following formula was used to calculate the density:

$$\frac{w}{((\Pi \times (d/2)2/2) - (d \times (d-t))) \times l}$$

The above calculation method takes into consideration slight asymmetries and irregularities in the core cutting method, the half-core not being an exact half-core due to the thickness of the blade and to the sometimes imperfect orientation of the blade against the core axis. A key assumption in the above density formula is that the sliver of rock cut by the diamond saw has straight sides so forming a quadrilateral polygon (Figure 4.57). In fact the two short sides will have a slight curve which are assumed in the above half core density equation to have straight sides so simplifying the half core volume section of the density formula. The error introduced by this assumption is deemed to not be significant in the overall volume, and hence density, calculation.

The difference between original calliper densities and recheck calliper densities was for most samples below 2.5% (10 negative and 10 positive). One sample (S229) showed a -4.7% difference.

The difference between the original calliper densities and the recheck saturated immersion densities was for most samples below 2.4% (7 negative and 12 positive). Sample S229 showed a -4.2% difference.

These checks confirm the reliability of the density database produced in 2021 (Figure 4.58). A repeat density measurement using the calliper method was undertaken on sample S229. It still showed a -4.7% difference with the original calliper measurement. This difference is deemed to be simply due to core heterogeneity as the original sample was a full core and the recheck sample a half core.

Figure 4.57 Schematic illustration of parameters used for volume calculation for half core originally cut to a cylindrical shape

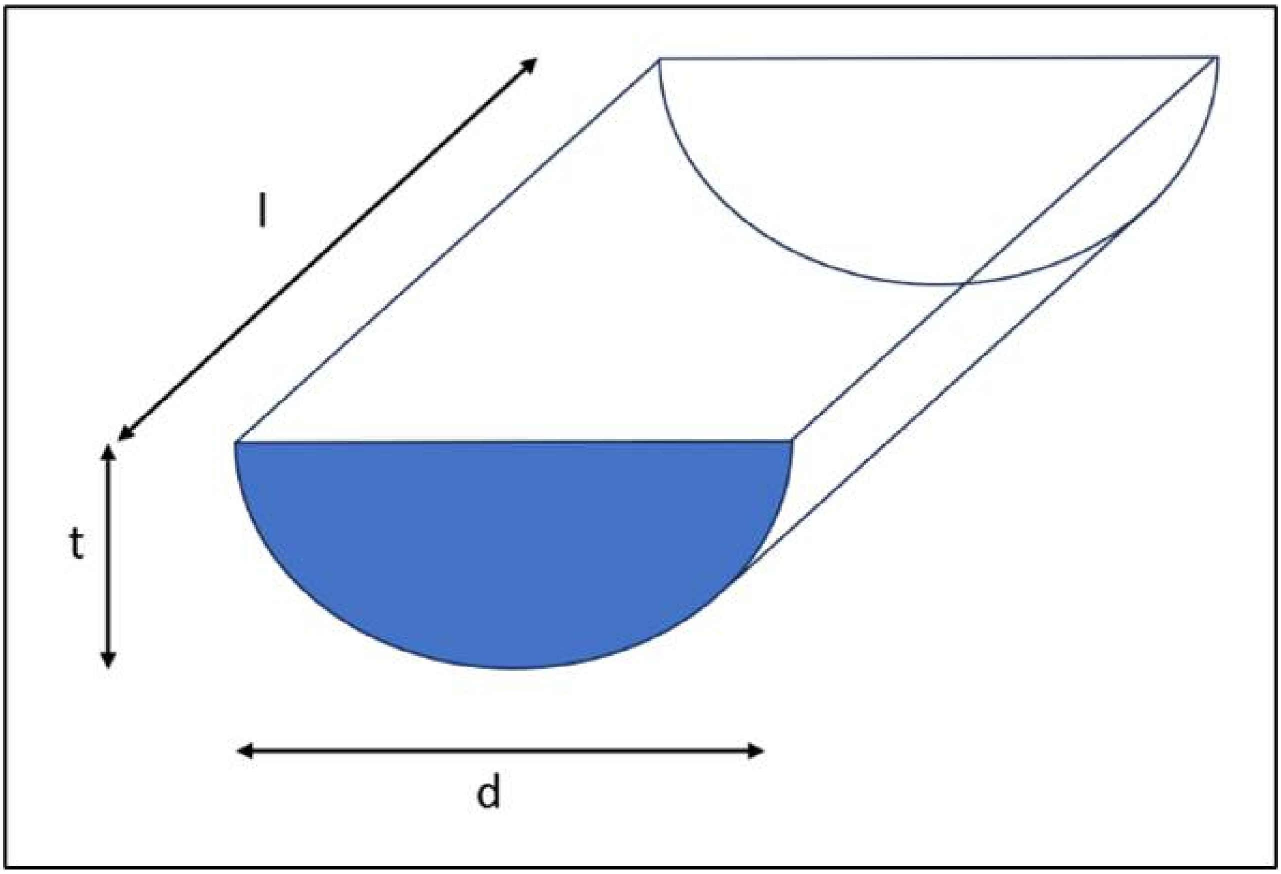
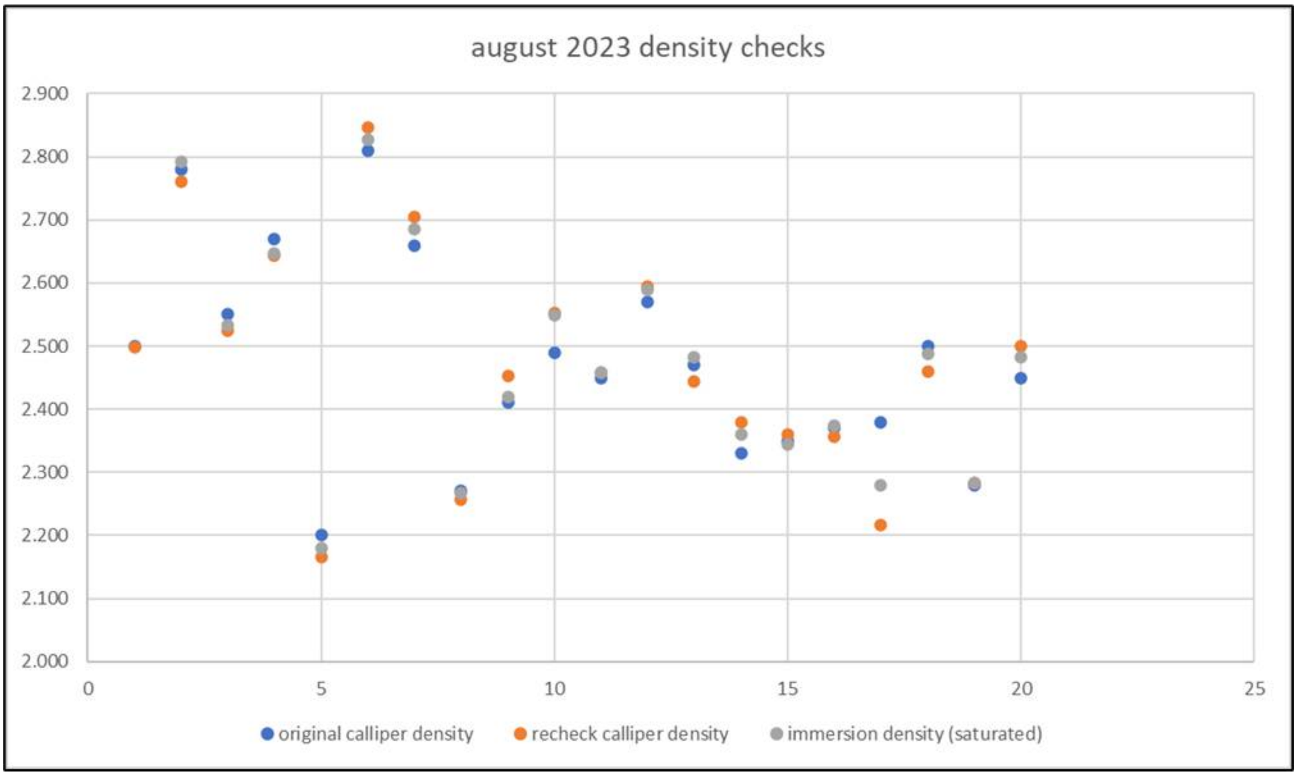


Figure 4.58 Density check measurements August 2023 summary



In the opinion of Snowden Optiro, the density checks conducted in August 2023 are suitable for estimation of Inferred and Indicated Mineral Resources. Snowden Optiro does, however, recommend that for future resource estimation further diamond core be obtained to measure density with appropriate QC procedures being applied.

Densities applied

The available density data was initially reviewed against the TREO grade, but there was no correlation between the two variables (Figure 4.59, Figure 4.60 and Figure 4.61).

The available density data was reviewed, and any low confidence density readings were excluded/ filtered. The mean naïve and length weighted density values were calculated, grouped by the interpreted weathering domain, grouped globally and the grouped by target area (Table 4.28). The final assigned density was assigned based on the available conditional mean (e.g. Target 4 has no density information and was assigned the Target 1 density values).

As there are no density measurements for material flagged as soil (WEATH=SO), and as a function of the limited volume this material represents (approximately 1.7% of the total mineralisation), an assumed density value of 1.8 t/ m^3 was assigned, which was derived by reducing the lowest measured density by 5%.

Figure 4.59 TREO – density grade relationship for fresh material types

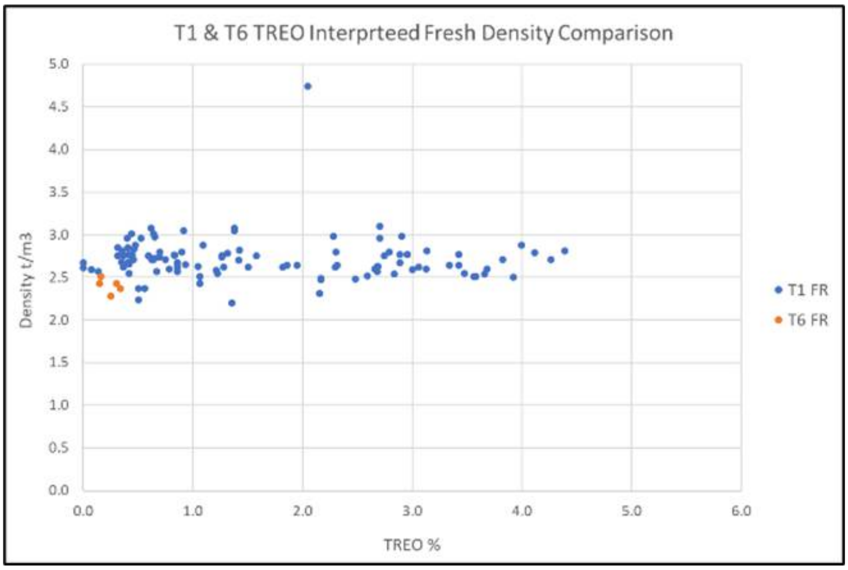


Figure 4.60 TREO – density grade relationship for slightly weathered material types

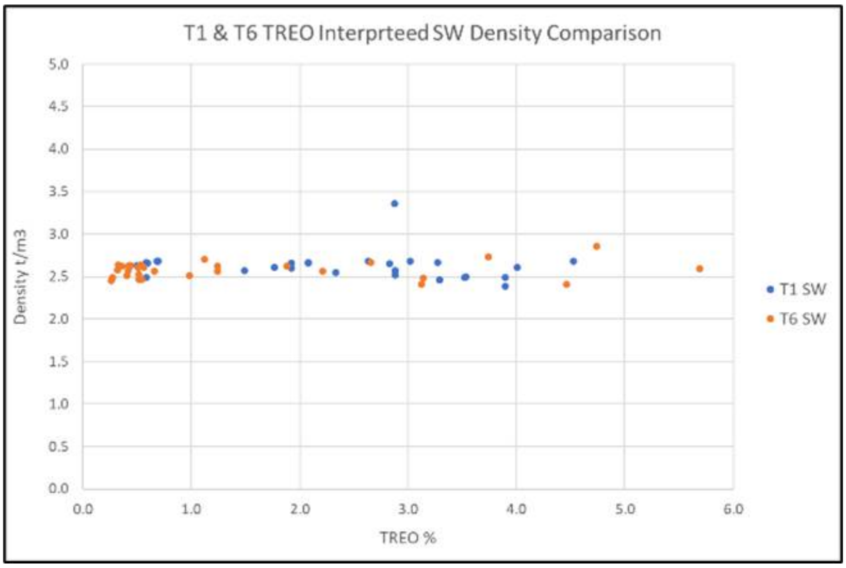


Figure 4.61 TREO – density grade relationship for weathered material types

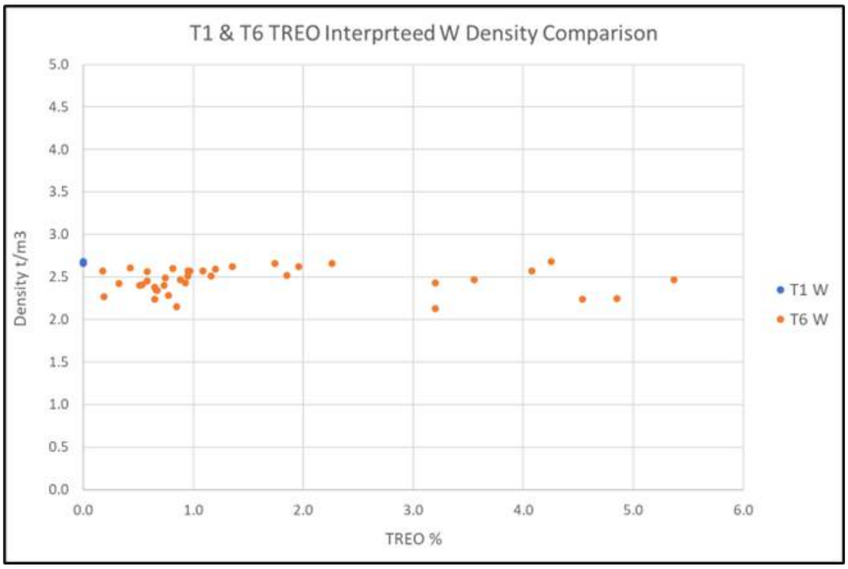


Table 4.28 Monte Muambe density data and model density values

Target	Interpreted weathering	Global filtered density data			Reported by target – filtered			Assigned density	
		No. of samples	Naïve average	Length-weighted average	No. of samples	Naïve average	Length-weighted average	Density (t/m^3)	Comment
1	SO							1.80	Assumed based on lowest measured density
	VW	23	2.12	2.08	5	2.69	2.68	2.10	Derived from global density
	W	40	2.47	2.46	2	2.67	2.67	2.55	Average of all density data
	SW	57	2.60	2.61	28	2.62	2.64	2.60	Derived from T1 density, rounded down
	FR	115	2.68	2.67	110	2.70	2.68	2.70	Derived from T1 density, rounded down
6	SO							1.80	Assumed based on lowest measured density
	VW	23	2.12	2.08	18	1.96	1.88	2.10	Derived from global density
	W	40	2.47	2.46	38	2.46	2.44	2.45	Derived from T6 density
	SW	57	2.60	2.61	29	2.58	2.58	2.60	Derived from global density
	FR	115	2.68	2.67	5	2.40	2.40	2.70	Derived from global density
4	SO							1.80	
	VW							2.10	
	W							2.55	All derived from T1 values
	SW							2.60	
	FR							2.70	

4.9 Mineral Resource classification

The Monte Muambe 2023 Mineral Resource has been classified and reported in accordance with the JORC Code (2012). The classification is based on the following:

- Confidence in the available geological and sample data.
- Confidence in the geological knowledge and interpretations.
- Confidence in the demonstrated geological and grade continuity.
- Confidence in the resultant Mineral Resource estimate.
- Spatial distribution of the available drill hole data.
- Satisfying the RPEEE requirements.

4.9.1 Joint Ore Reserve Committee Reporting Code (2012 edition)

The current JORC Code (2012) is an internationally recognised guideline for the reporting of Mineral Resources and Ore Reserves. The JORC Code (2012) provides specific definitions. Clause 20 of the reporting guidelines outlines the criteria for reporting a Mineral Resources with the key considerations being:

- The material of interest of economic interest is in such form, grade (or quality) and quantity that there are reasonable prospects for eventual economic extraction, regardless of the classification. Portions of a deposit that do not meet the RPEEE criteria, must not be included in a Mineral Resource.
- The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated, or interpreted. This relates to the confidence in the available data, knowledge of the deposit as well as the confidence in the MRE. This will determine the MRE classification.

4.9.2 Monte Muambe September 2023 model

The mineralisation at Monte Muambe is defined by a 0.5 to 1.0% TREO domain, and a ≥1.0% TREO domains. The grade criteria reflect what is currently understood about the mineralisation and do not reflect any economic consideration – this approach is broadly considered best practice. Although there is a reasonable understanding of the geology, understanding the geology is an on-going process and is always a function of the available data.

In the broader Target 1 area, there are multiple mineralised intersections and positions, of varying confidence. Two options exist when preparing the mineralisation interpretation. Mineralised positions (i.e. external to the main 0.5% TREO mineralisation) could be ignored and not interpreted/ modelled, or they can be included and then the confidence/classification field used to assist in ranking their value to the Project, which was carried out. However, this results in a volume of mineralisation which currently has little opportunity of being classified as a Mineral Resource.

Confidence/resource classification

Classification of the 2023 Monte Muambe Mineral Resource was a multi-stage process:

- The data was reviewed both during loading and desurveying of the data as well as during the construction of the interpretations. Importantly construction of the wireframes informs the understanding spatial distribution of data and the confidence in the interpretations in a 3D space.
- The sample data was flagged by the estimation domain (domain) and weathering flags, which was the basis for subsequent geological/statistical analysis and variography (variography is a spatial statistic informing the continuity of grade).
- Post-estimation several estimation metrics are reviewed to assess the quality of the estimate. These metrics include:
  - Number of informing samples and drill holes

  - Kriging variance (a measure of the quality of the relationship between the modelled variogram and the spatial arrangement of data)
  - Search pass and distance to nearest sample
  - Kriging efficiency and slope of regression – two summary variables measuring the quality of the estimate.
- The informing drill holes in combination with the various estimation metrics were used to delineate area of moderate, low, and very low confidence (CONFID), but which has no RPEEE consideration. The CONFID field is coded into the block model.
- The open pit optimisation is then run using the model. This is used to specify material that it supports positive cash flows from mining and processing.
- All blocks that are outside of the optimised pit shell are flagged as being unclassified (UNCL) – failing to meet the RPEEE criteria. For blocks inside the pit shell (and having met the RPEEE criteria) are then classified with the RESCAT field:
  - Any MOD confidence blocks are flagged as Indicated (IND)
  - Any LOW confidence blocks are flagged as Inferred (INF)
  - Any VLOW confidence blocks are flagged as unclassified (UNCL).

The Monte Muambe 2023 Mineral Resource has been classified and reported in accordance with the JORC Code (2012). The classification reflects:

- Confidence in the available geological and sample data.
- Confidence in the geological knowledge and interpretations.
- Confidence in the demonstrated geological and grade continuity.
- Confidence in the resultant Mineral Resource estimate.
- Spatial distribution of the available drill hole data.
- Results of the open pit optimisation.
- Finally with the reporting of the Mineral Resource, a grade cut-off of 1.5% TREO is applied to reflect material that will meet the RPEEE criteria.

Applying the confidence and economic consideration, blocks within the pit shells informed with a minimum drill hole spacing approaching 80 m along strike by 80 m across strike, in addition to demonstrated geological and grade continuity, and where the grade has been extrapolated no more than 35 m across strike, was classified as an Indicated Mineral Resource. Where informed by wider spaced drilling and/ or where geological and grade continuity was assumed, the Mineral Resource was classified as an Inferred Mineral Resource. The grade – tonnage tables for Target 1 and Target 4 are shown in Table 4.29 and Table 4.30 below.

Table 4.29 Grade tonnage table for Target 1

Target	CONFID	TREO cut (%)	Volume (Mt)	TREO %	Ce_2O_3	Dy2O_3	Nd_2O_3	Pr_2O_3	Tb_2O_3	Nb_2O_5	Th	U	CeO_2	Pr_6O_{11}	Tb_4O_7
		0.00	28.89	1.24	5,535	74	1,410	491	14	1,040	159	18	5,805	507	14
		0.25	28.89	1.24	5,535	74	1,410	491	14	1,040	159	18	5,805	507	14
		0.50	28.87	1.24	5,537	74	1,410	492	14	1,039	159	18	5,807	507	14
		0.75	15.21	1.75	7,943	77	1,805	670	15	830	177	19	8,330	692	15
		1.00	9.91	2.24	10,256	82	2,170	836	16	728	195	20	10,756	863	15
		1.25	9.10	2.34	10,725	81	2,228	867	15	623	197	20	11,248	895	15
1	**Indicated & Inferred**	**1.50**	**8.82**	**2.37**	**10,855**	**80**	**2,244**	**876**	**15**	**592**	**196**	**20**	**11,384**	**905**	**15**
		1.75	8.20	2.43	11,064	80	2,277	895	15	559	196	20	11,603	924	15
		2.00	6.87	2.54	11,449	80	2,352	934	15	530	199	21	12,007	964	15
		2.25	5.13	2.67	11,957	82	2,462	987	16	510	203	22	12,540	1,019	16
		2.50	3.51	2.81	12,424	83	2,561	1,036	16	490	207	23	13,029	1,070	16
		2.75	1.73	3.01	13,048	84	2,698	1,103	16	461	211	23	13,684	1,139	16
		3.00	0.67	3.24	13,688	90	2,899	1,194	18	484	226	25	14,356	1,232	18

Table 4.30 Grade tonnage table for Target 4

Target	CONFID	TREO cut (%)	Volume (Mt)	TREO %	Ce_2O_3	Dy_2O_3	Nd_2O_3	$P_{r2}O_3$	Tb_2O_3	Nb_2O_5	Th	U	CeO_2	Pr_6O_{11}	Tb_4O_7
		0.00	16.32	1.34	5,519	139	1,371	473	25	979	367	12	5,788	488	25
		0.25	16.32	1.34	5,519	139	1,371	473	25	979	367	12	5,788	488	25
		0.50	16.32	1.34	5,519	139	1,371	473	25	979	367	12	5,788	488	25
		0.75	12.62	1.53	6,396	142	1,510	534	26	910	353	12	6,707	552	26
		1.00	7.27	2.03	8,624	142	1,869	695	26	712	344	12	9,044	718	26
		1.25	5.10	2.43	10,443	144	2,151	824	26	508	334	7	10,952	850	26
4	**Inferred**	**1.50**	**4.80**	**2.50**	**10,734**	**143**	**2,189**	**843**	**26**	**481**	**330**	**7**	**11,258**	**870**	**26**
		1.75	3.85	2.71	11,678	140	2,298	902	25	420	319	4	12,247	931	25
		2.00	2.88	3.00	12,984	134	2,436	979	24	336	298	1	13,616	1,010	24
		2.25	2.53	3.12	13,552	134	2,515	1,016	24	304	296	1	14,212	1,049	24
		2.50	2.09	3.28	14,286	137	2,644	1,070	25	280	302	0	14,983	1,104	25
		2.75	1.55	3.51	15,358	148	2,855	1,153	27	270	324	0	16,107	1,190	27
		3.00	1.36	3.60	15,790	150	2,934	1,185	27	262	331	0	16,559	1,224	27

Cut-off grade considerations

A grade cut-off of 1.5% TREO was applied to the reporting of the Mineral Resource to reflect the likely/ expected profitable material to be mined. There are multiple options to identify material that can be profitably mined and hence dictate the reporting cut-off including:

- How the mineralisation presents (i.e., can a spatially consistent block be identified for mining).
- Likely mining and processing costs as well as revenue considerations.
- Corporate considerations including process and mining rates, financing decisions and governance criteria. Options including stockpile strategies are then available to maximise the value of the operation.

Many of the grade considerations are the result of numerous scenarios and development iterations of the available parameters.

4.9.3 Depletion of resource model

The Monte Muambe 2023 maiden Mineral Resources have not been previously mined and there is no depletion.

4.9.4 Comparison to previous estimates

As a maiden Mineral Resource, there are no previous estimates for the Target 1 and Target 4 Mineral Resource.

4.10 Mineral Resource statement

The Mineral Resource Estimate is provided in Table 4.31. The Mineral Resource is reported in accordance with the JORC Code (2012).

Table 4.31 Monte Muambe Indicated and Inferred Mineral Resource September 2023 reported using a 1.5% TREO cut-off

Target	Classification	TREO cut-off (%)	Tonnes (Mt)	TREO (%)	CeO_2 (ppm)	Pr_6O_{11} (ppm)	Nd_2O_3 (ppm)	Tb_4O_7 (ppm)	Dy_2O_3 (ppm)	NdPr Oxide (ppm)	Contained TREO (t)
1	Indicated	1.5	8.0	2.38	11,400	910	2,250	15	80	3,160	191,000
	Inferred	1.5	0.8	2.28	10,900	861	2,140	15	78	3,000	18,000
	TOTAL	**1.5**	**8.8**	**2.38**	**11,400**	**905**	**2,240**	**15**	**80**	**3,150**	**209,000**
4	Indicated	1.5									
	Inferred	1.5	4.8	2.50	11,300	872	2,190	26	143	3,060	119,000
	TOTAL	**1.5**	**4.8**	**2.50**	**11,300**	**872**	**2,190**	**26**	**143**	**3,060**	**119,000**
OVERALL	Indicated	1.5	8.0	2.38	11,400	910	2,250	15	80	3,160	191,000
	Inferred	1.5	5.6	2.47	11,200	871	2,190	24	134	3,060	137,000
	TOTAL	**1.5**	**13.6**	**2.42**	**11,400**	**894**	**2,230**	**19**	**102**	**3,120**	**329,000**

Notes:

- *Million tonnes are rounded to one decimal place. Grades are rounded to two decimal places for % and whole numbers for ppm.*
- *The MRE has been reported in consideration of reasonable prospects for eventual economic extraction (RPEEE) using a pit shell based on a 1.5% Total Rare Earth Oxide (TREO) cut-off, revenue of 24.65 $/kg TREO in Mixed Rare Earth Carbonate (MREC) and average total recovery to MREC of 48%.*
- *Mineral Resources are reported as dry tonnes on an in-situ basis.*
- *Rare Earth Elements are inclusive of the TREO and not additional to it.*
- *"NdPr Oxide" is the sum of Nd_2O_3 and Pr_6O_{11}.*

4.10.1 Reasonable prospects for eventual economic extraction

RPEEE was applied through the constraint of an optimised pit shell. The optimised pit shell was developed using parameters developed for the project and provided by Altona to Snowden Optiro. Where direct parameters have not been developed, benchmarking from similar projects has been used. The TREO price used is $24,651/ t. The inputs used for the pit optimisation are shown in Table 4.32.

Table 4.32 September 2023 Mineral Resources reported at a cut-off grade of 1.5% TREO

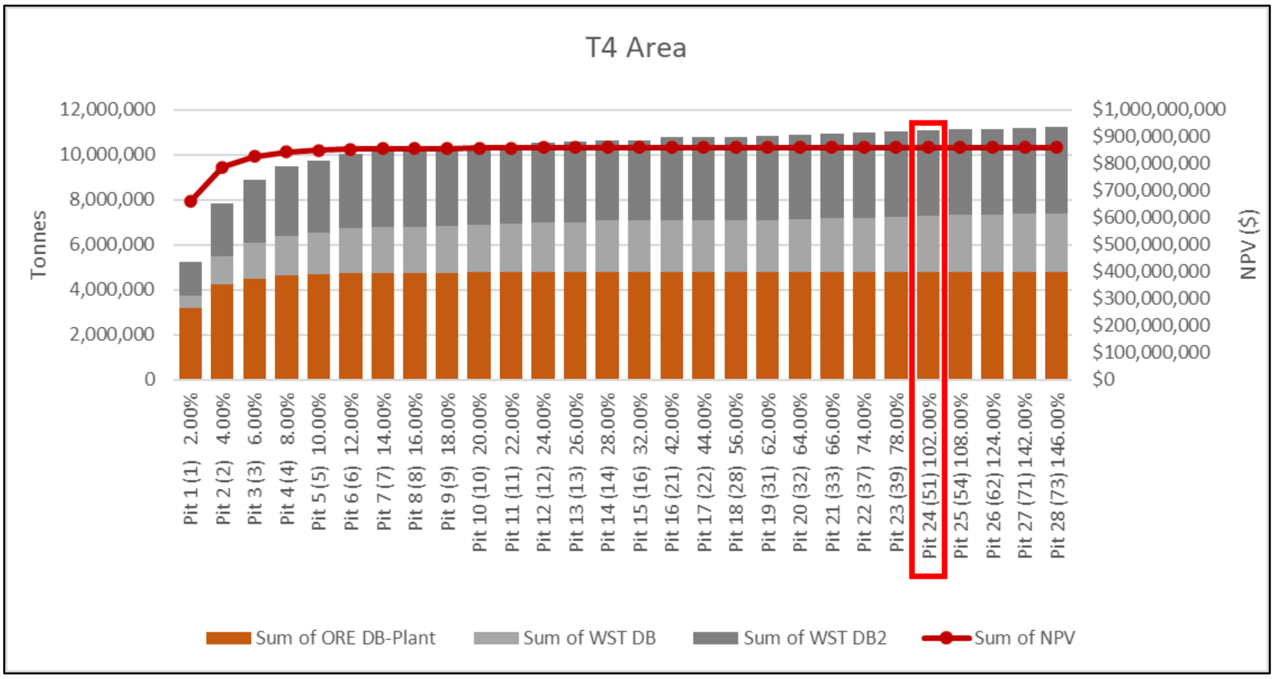
Price	Value	Unit
TREO	24 651	$/t TREO in MREC
Royalties	3	%
Mining costs		
Ore – Free dig	3.28	$/t
Ore – Drill and blast	4.26	$/t
Waste – Free dig	2.51	$/t
Waste – Drill and blast	3.53	$/t
Process cost	**25**	**$/t of ROM**
Other ore costs (G&A and incremental ore mining)	**66**	**$/t of TREO**
Recovery		
Process recovery	60	% Recovery
Hydrometallurgical plant recovery	80	% H Plant
Ore production	**750,000**	**t p/a**
Discount rate	**10**	**%**
Slope angles		
T4 overall angle	47	
T1 T6 overall angle	43	

The estimation of the TREO final price was based on the long-term prices for the contained products, praseodymium, neodymium, terbium and dysprosium and the proportion they report on the resource model.

The cut-off grade of 1.5% of TREO was agreed with the Client, based on the grade and tonnage curve. Also, the CONFID attribute defined which confidence of material would be used for the optimization: MOD and LOW. These are equivalent to Indicated and Inferred classification respectively. Two distinct mining approaches were delineated based on rock competency, namely free dig and drill and blast. Given the limited precision of lithology data at this stage, materials categorized as ‘soft’ (roughly equivalent to soil cover) would be allocated to the free dig operation, while all other materials would undergo the drill and blast method.

The resource model was divided into the two different areas, one containing Target 1 and 6, and another for Target 4. The ultimate pit results are also split between them. For Target 4 area the pit shell chosen was using the price increment of 102% which provided the highest NPV for the case. For Target 1 and 6, the chosen price increment was 98% which also provided the highest NPV result (Figure 4.62 and Table 4.33).

Figure 4.62 NPV graphs for all target areas



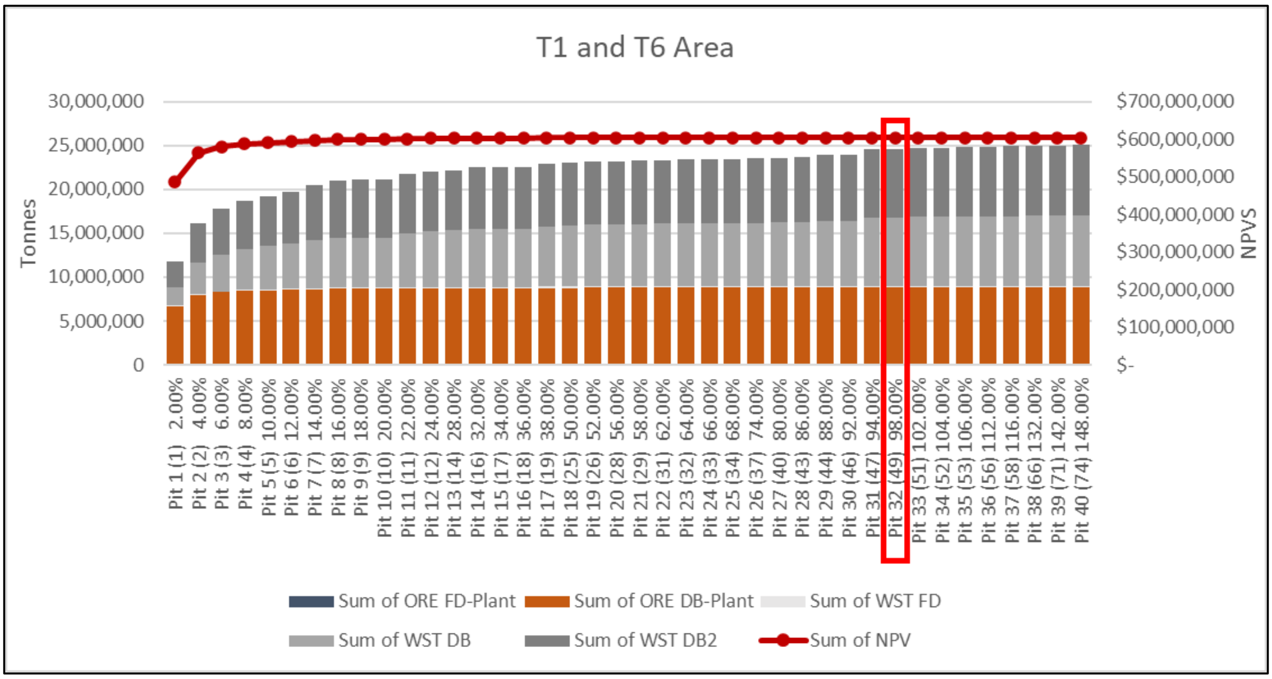


Table 4.33 Ore and waste tonnages for the pits alongside calculated NPV

Deposit	Ore (t)	Waste (t)	Net present value ($ M)
T1 T6	8.81	15.77	603.87
T4	4.80	6.27	860.77
Total	**13.60**	**22.04**	**1,464.64**

Note: Totals may not add up due to rounding

4.11 Independent reviews

No independent reviews have been undertaken on the mineral resources, as this is the maiden estimation.

4.12 Snowden Optiro comments on Mineral Resource estimate

The Monte Muambe 2023 Mineral Resource estimate is a maiden Mineral Resource and represents the first 3D integration of the available geology and data. The estimate was derived from appropriate data, that was used to develop the TREO grade based mineralised interpretations: a low grade 0.5–1.0% TREO and a high grade ≥1.0% TREO interpretations. The lower grade mineralisation appears as a dispersion halo, surrounding the higher-grade mineralisation, with the implication that the weathering/oxidation has created the halo around the higher grade. In addition, there is some preliminary evidence that the carbonatite is fractionated/ zoned, but an understanding of the nature and impact of this fractionation is still being developed, and no material impact has been identified to date. Within the mineralised envelopes, the grades exhibit low variability. No discrepancies were identified during validation, and there is reasonable confidence that the estimated grade suitably reflects the available geological understanding and sampling.

The Mineral Resource has been reported in accordance with the JORC Code (2012) reporting guidelines and has been constrained by an optimised pit shell to reflect the reasonable prospects for eventual economic extraction, The Mineral Resource employed a range of metrics to reflect the overall confidence in the estimate. Target 1 has been tested with regularly spaced data and the geological and grade continuity can be assumed, and the data is sufficient to define Indicated and Inferred Mineral Resources. At Target 4, the drill hole spacing is sufficient to imply geological and grade continuity and has been classified as an Inferred Mineral Resource.

There remains a significant amount of mineralisation that has been tested to varying extents, but which remains a mineralised inventory. This material is poorly informed/defined and either lacks sufficient confidence in the interpretation or sufficient sampling, and/or is outside of the optimised pit shell. It is expected that with on-going exploration, some of this mineralised inventory could be converted to a Mineral Resource.

On-going drilling and geological work is required to continue to develop the spatial geological understanding of the deposit. This includes improved understanding of the deposit geochemistry, weathering/oxidation features as well as collecting additional bulk density samples.

5 MINING

The mining method is based on conventional open pit using truck and shovel, and drill and blast, coupled to a ROM stockpile. Although the rock is largely classified as weathered, ore and waste rock will require drilling and blasting.

Both ore and waste will be excavated in 5 m flitches following mark-out by grade control. Ore will be hauled to either the ROM pad and tipped onto a designated ore finger or a designated low-grade stockpile. All mine waste will be hauled directly from the pit and placed onto a designated location of the tailings storage facility (TSF) dam wall; there are no other external waste dumps.

The mining fleet will comprise 40 – 60 t capacity articulated dump trucks (such as a Caterpillar 745) loaded by a 90-t excavator (such as a Caterpillar 395). A 30-t front-end loader (Caterpillar 980M) capable of loading the 41-t dump trucks, will be used as back-up for the primary loading unit and to make up shortfalls in periods where additional material movement is required. Other ancillary support will be supplied by a Cat D9R dozer, Cat 14M grader, and Cat 745 watercart. Maintenance will be conducted on site. Contract-mining is selected as the operating strategy at the Project.

5.1 Pit optimisation

Optimization parameters used for the Scoping Study are summarised in Table 5.1. Parameters were sourced as follows:

- TREO price and royalty – Altona. The estimation of the TREO final price was based on the long-term prices for the containing products, praseodymium, neodymium, terbium and dysprosium and the proportion they report on the resource model.
- Mining operating costs – assumed by Snowden Optiro based on pricing reported for a nearby rare earths operation of a similar scale.
- Processing costs, recovery assumptions and discount rates – Altona.
- Overall open slope angles (OSA) – assumed by Snowden Optiro based on a consideration of likely rock mass strength and the proportion of oxide and transition rocks in the top of the pit.

Table 5.1 Parameters used in optimization

Item	Unit	Value
TREO	$/t	24,651
Royalty	%	3.0
Mining costs	$/t	3.28
Ore – free dig	$/t	4.26
Ore – drill and blast	$/t	2.51
Waste – free dig	$/t	3.53
Waste – drill and blast	$/t	3.28
Processing cost	$/t ore	25.00
Downstream processing cost	$/t TREO	66.00
Recovery from ROM	%	60
Recovery from refining	%	80
Throughput rate	Tonnes per year	750,000
Discount rate	%	10
Overall slope angle (OSA) T4	°	47
OSA T1 and 6	°	43

5.1.1 Dilution and ore loss

A large proportion of the Project mineralisation has lateral widths of more than 30 m and mineralisation that is visually distinguishable by colour from the adjacent fresh rock, thereby providing ease of selectivity and minimal overall effect from ore loss at edges. No dilution was added to the block model for optimisation, however a 5% dilution at zero grade, and a 5% ore loss has been applied in the mining schedule.

5.1.2 Geotechnical parameters

No geotechnical studies have been undertaken. Typical bench height and berm widths with OSA are weathered (21.3°), mixed (27.6°) and fresh (38.8°).

The groundwater depth has not yet been determined; but it is generally assumed that the water table is generally more than 100 m below surface and in many areas not encountered in drill holes. Mine dewatering is expected to be undertaken using in-pit drainage and sumps, with contingency measures for unexpected pit inflows.

5.1.3 Classification

The CONFID attribute defined which confidence of material would be used for the optimization. All resource confidence classifications were included in the optimisation. There is no Measured or Indicated classification in the block model. The confidence in the mining schedules is a function of the low confidence of the resource estimate. A cut-off grade of 1.5% of TREO was agreed, based on the grade and tonnage curve.

5.1.4 Optimisation results

For Target 4 area the pit shell chosen was using the price increment of 102% which provided the highest undiscounted NPV as summarised in Figure 5.1. For Target 1 and 6, the chosen price increment was 98% which provided the highest undiscounted NPV as shown in Figure 5.2.

Figure 5.1 T4 optimisation result

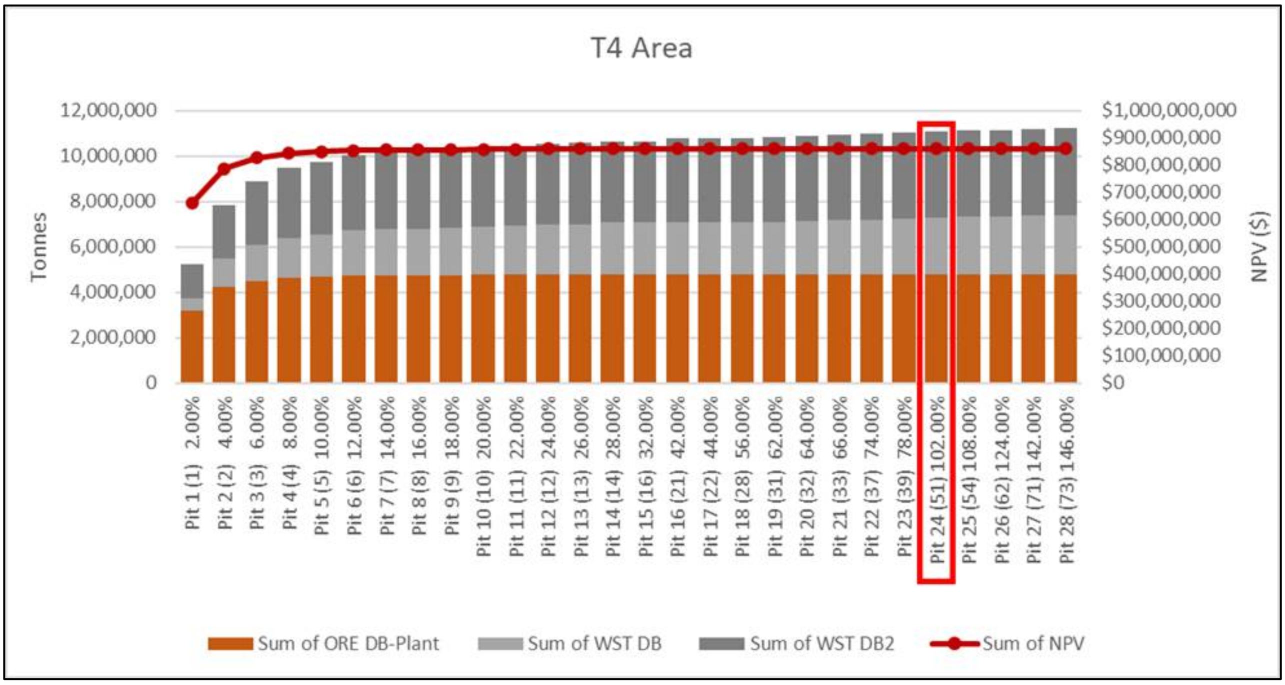
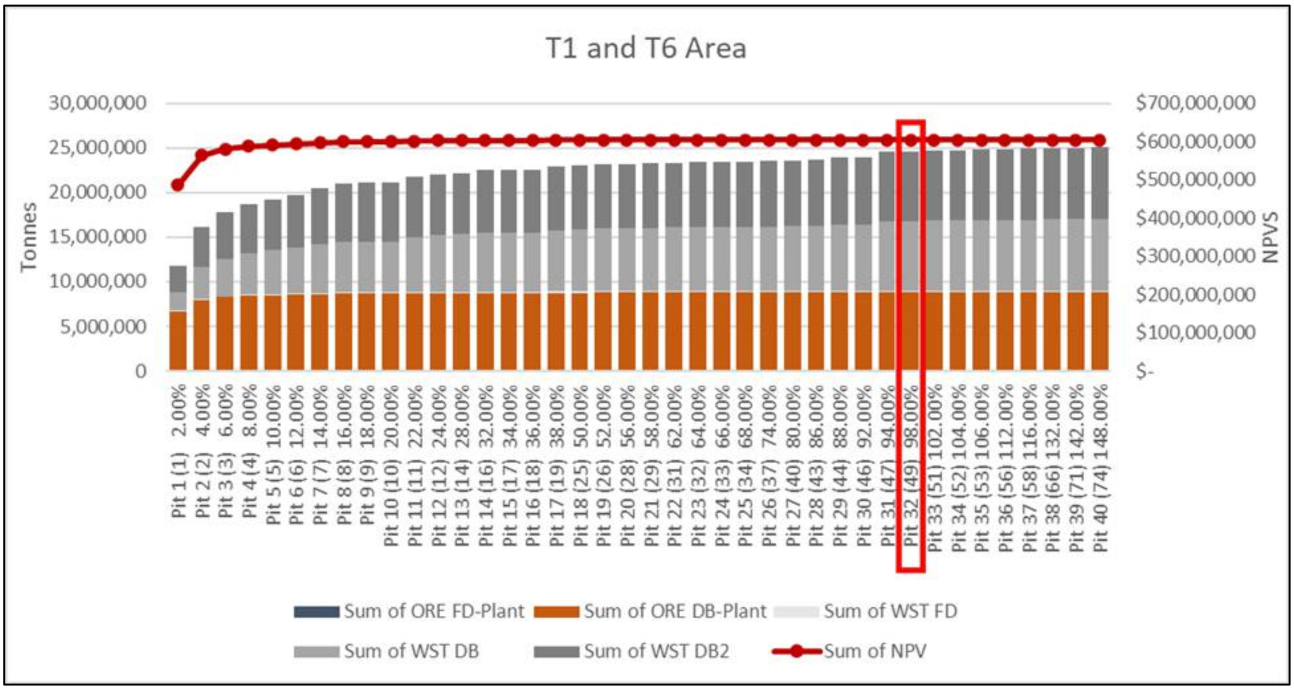


Figure 5.2 T 1 and 6 optimisation result



5.2 Mine design

The pit and stage designs were based on the slope parameters described above, using ramp widths of 10 m. All mine waste rock will be dumped external to the pit and used for the construction of the walls for the TSF. This has eliminated the need for an external rock dump. Any low-grade ore stockpiles will be reclaimed and processed in the final years of the schedule. In the planned mine schedule, all material <1.5% TREO has been classified as waste.

The mine life is planned at 18 years. There is no pre-strip period. Based on the selected pit shells a high-level pit design was produced based on the following parameters:

- Face angle of 60°.
- Berm width of 3.7 m.
- Ramp width of 10 m and 10% gradient

Indicative pit designs are shown in Figure 5.3 and Figure 5.4. Table 5.2 compares the pit optimisation ore and waste tonnages with the design. Differences are within normal tolerances.

Figure 5.3 T2 and T6 pit design

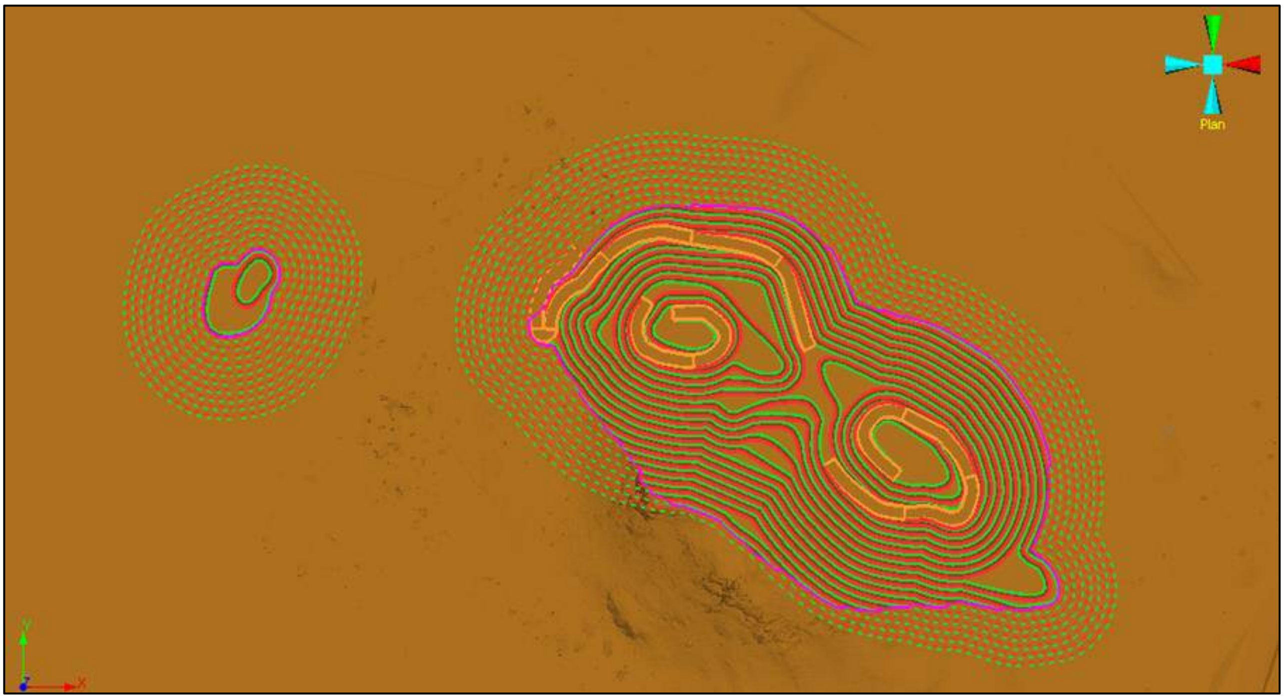


Figure 5.4 T1 pit design

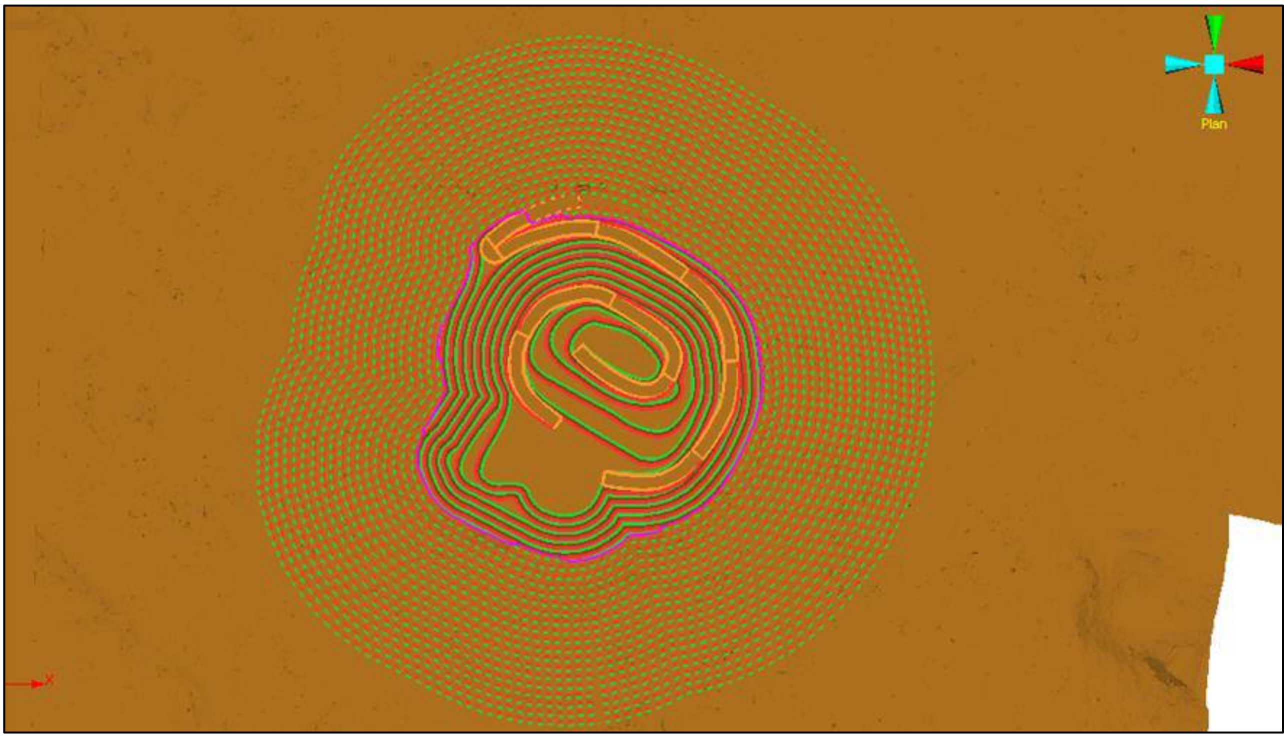


Table 5.2 Comparison of optimisation and pit design tonnages

Area	Unit	Optimisation	Design
T4			
Ore	Mt	4.799	4.588
Waste	Mt	6.430	6.565
Total T4	**Mt**	**11.229**	**11.153**
T 1 and 6			
Ore	Mt	8.805	8.568

Area	Unit	Optimisation	Design
Waste	Mt	15.766	15.274
Total T1 and 6	**Mt**	**24.572**	**23.842**

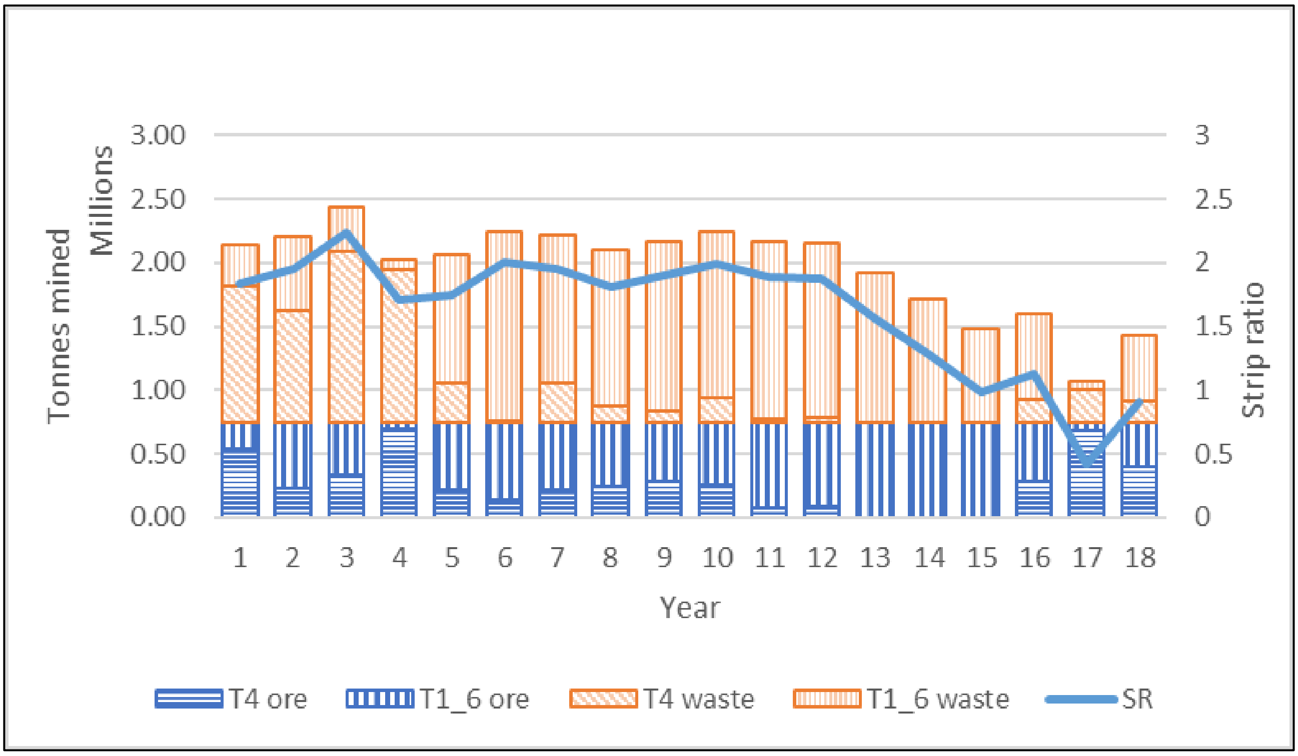
5.3 Ore Reserve estimate

The technical study is not to a pre-feasibility level. Consequently, no Ore Reserve was estimated or reported.

5.4 Indicative schedule

An indicative life of mine (LOM) schedule was prepared for the mining of the two open pits as shown in Figure 5.5. Throughput rate is maintained at 750,000 t/a at a total mining rate of between 2.0 to 2.5 Mt/a. The average strip ratio is 1.67 (waste: ore). A pre strip period is not required but may be used to generate sufficient waste for the first TSF lift.

Figure 5.5 Monte Muambe annual LOM mining schedule



5.5 Processing schedule

An annualised processing schedule is summarised in Table 5.3 (Year 1 - 9) and Table 5.4 (Year 10 – 18).

Table 5.3 LOM annualised processing schedule (Y1 – Y9)

Parameter	Unit	1	2	3	4	5	6	7	8	9
Run of mine tonnes	Mt	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
TREO	%	2.82	2.43	2.51	2.72	2.51	2.47	2.62	2.34	2.56
DY_2O_3	%	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
ND_2O_3	%	0.29	0.26	0.24	0.23	0.24	0.24	0.24	0.22	0.22
NB_2O_5	%	0.08	0.09	0.07	0.05	0.07	0.07	0.06	0.06	0.05
PR_2O_3	%	0.10	0.10	0.09	0.09	0.09	0.09	0.09	0.08	0.09
TB_2O_3	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
CEO_2	%	1.26	1.13	1.13	1.18	1.12	1.09	1.16	1.03	1.12
PR_6O_{11}	%	0.11	0.10	0.10	0.09	0.09	0.09	0.10	0.09	0.09
TB_4O_7	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Table 5.4 LOM annualised processing schedule (Y10 – Y18)

Parameter	Unit	10	11	12	13	14	15	16	17	18	Total
Run of mine tonnes	Mt	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	**13.50**
TREO	%	2.43	2.45	2.32	2.39	2.37	2.39	2.27	2.09	1.95	**2.42**
DY_2O_3	%	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	**0.01**
ND_2O_3	%	0.22	0.23	0.21	0.21	0.21	0.20	0.19	0.19	0.18	**0.22**
NB_2O_5	%	0.05	0.05	0.05	0.04	0.04	0.04	0.03	0.04	0.04	**0.06**
PR_2O_3	%	0.09	0.09	0.08	0.09	0.08	0.08	0.08	0.07	0.07	**0.09**
TB_2O_3	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**0.00**
CEO_2	%	1.09	1.12	1.05	1.10	1.09	1.08	1.00	0.89	0.85	**1.08**
PR_6O_{11}	%	0.09	0.09	0.09	0.09	0.09	0.08	0.08	0.07	0.07	**0.09**
TB_4O_7	%	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**0.00**

6 METALLURGY AND PROCESSING

6.1 Metallurgical testwork

6.1.1 Sample description

Two bulk sample composites were generated (Table 6.1) from RC cuttings:

- Low-grade (LG) composite.
- High-grade (HG) composite.

Both composites were homogenised and stage ground to 80% passing 150 microns. A head grade for each sample (Table 6.2) was determined to be:

- 1.05% TREO for the LG composite.
- 3.22% TREO for the HG composite.

Assay results for the composites are shown in Table 6.2.

Table 6.1 Composited intervals for metallurgical samples

Low grade sample (LG)						High grade sample (HG				
Hole	From	To	Weight (kg)	Tag No		Hole	From	To	Weight (kg)	Tag No
MM073	13	14	1	L6850		MM074	28	29	1	L6856
MM073	14	15	1			MM074	30	31	1	
MM073	15	16	0.95			MM074	31	32	1	
MM073	16	17	1			MM074	32	33	1.05	
MM073	17	18	1.05			MM074	33	34	1	
MM073	18	19	1.1	L6851		MM074	34	35	1	L6858
MM073	19	20	1.025			MM074	36	37	1	
MM073	22	23	1			MM074	37	38	1.05	
MM073	23	24	1			MM074	38	39	1	
MM073	24	25	0.975			MM074	39	40	1	
MM073	25	26	1	L6852		MM093	14	15	1	L6859
MM073	26	27	1			MM093	15	16	1	
MM073	27	28	1.05			MM093	16	17	1	
MM073	28	29	1			MM093	17	18	1	
MM073	29	30	0.95			MM093	18	19	1	
MM073	30	31	1	L6853		MM093	19	20	1	L6860
MM073	31	32	1.05			MM093	20	21	1	
MM073	32	33	1.05			MM093	21	22	1	
MM073	33	34	1			MM093	22	23	0.95	
MM073	34	35	0.95			MM093	23	24	1	
MM061	7	8	0.95	L6854		MM093	47	48	1	L6861
MM061	8	9	0.95			MM093	48	49	1	
MM061	9	10	1.025			MM093	49	50	1	
MM061	10	11	1			MM093	50	51	1	
MM061	11	12	1			MM093	51	52	1	
MM061	12	13	1	L6855		MM093	52	53	1	L6862
MM061	13	14	0.95			MM093	53	54	1	
MM061	14	15	1.05			MM093	54	55	0.95	
MM061	15	16	1			MM093	55	56	1	
MM061	16	17	1			MM093	56	57	1	
Total weight			**30.1 kg**			**Total weight**			**30.0 kg**	

Table 6.2 Assay results for composites

Element	Units	HG Comp	LG Comp
TREO	**%**	**3.22**	**1.05**
NdPrO	**%**	**0.41**	**0.22**
La_2O_3	ppm	11,728	2,580
CeO_2	ppm	15,232	4,913
Pr_6O_{11}	ppm	1,208	604
Nd_2O_3	ppm	2,916	1,633
Sm_2O_3	ppm	232	58
Eu_2O_3	ppm	58	58
Gd_2O_3	ppm	115	58
Tb_4O_7	ppm	17	20
Dy_2O_3	ppm	87	100
Ho_2O_3	ppm	16	18
Er_2O_3	ppm	41	47
Tm_2O_3	ppm	5	5
Yb_2O_3	ppm	46	34
Lu_2O_3	ppm	4.5	5.0
Y_2O_3	ppm	508	381
U	ppm	<10	<10
Th	ppm	270	180
K_2O	%	0.17	1.47
Fe_2O_3	%	26.4	18.9
P_2O_5	%	2.3	4.7
Al_2O_3	%	1.04	2.76
CaO	%	25.5	30.7
SiO_2	%	2.8	10.0
MgO	%	2.52	0.77
BaO	%	1.50	0.94
MnO	%	7.20	4.71
SO_3	%	0.21	0.06
Nb	%	0.05	0.19
Sc	ppm	16	6
Zr	%	0.02	0.04
LOI1000	%	18.16	17.78

6.1.2 Beneficiation testwork

Size and assay deportment

The samples were screened, and the size fractions assayed for rare earths and typical gangue elements. There were no significant biases (upgrade or rejection) of either the rare earths or gangue elements by mass deportment. The results are shown graphically in Figure 6.1 and Figure 6.2.

Figure 6.1 LG composite size fractions

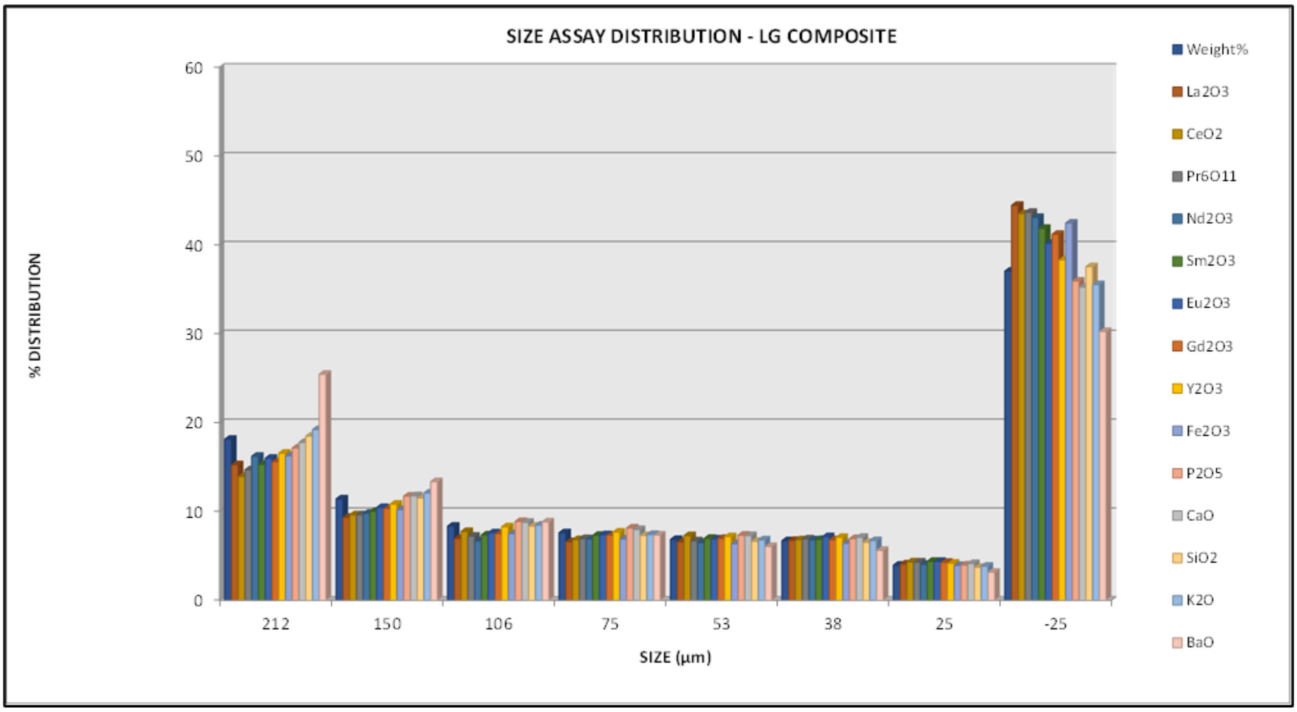
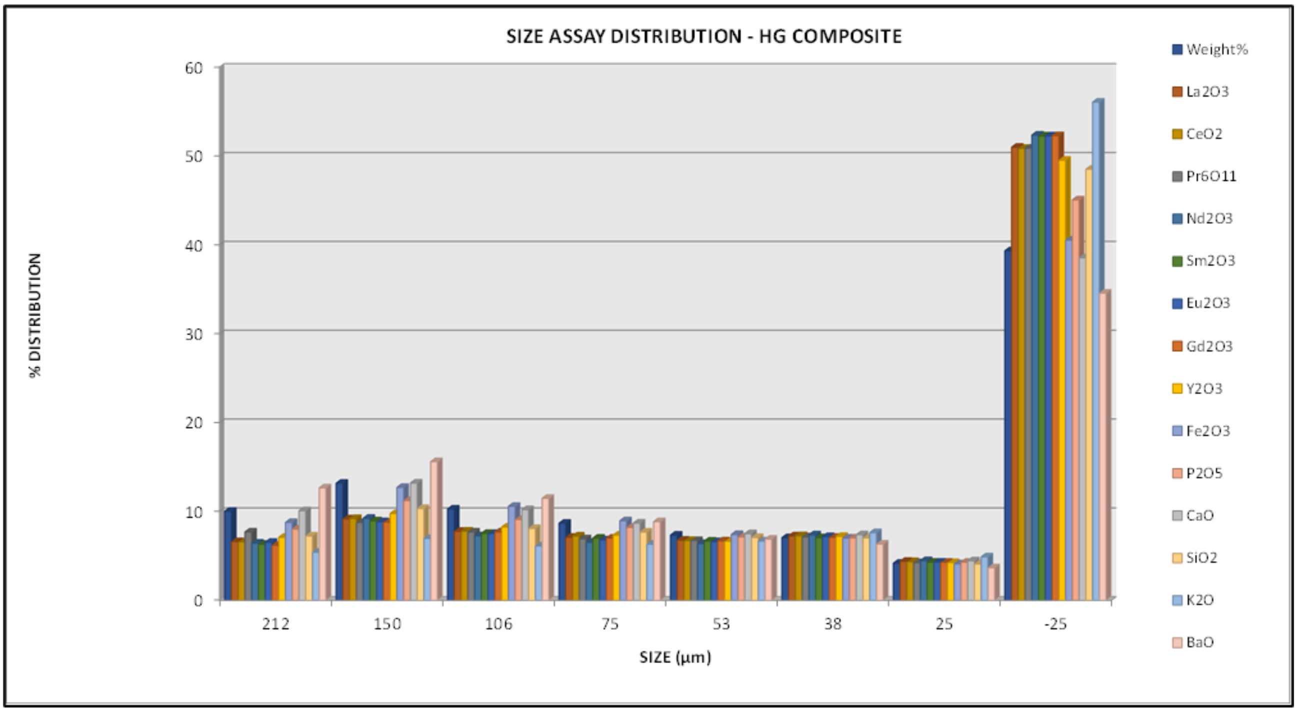


Figure 6.2 HG composite size fractions



Magnetic separation

The LG and HG composites were subjected to sighter wet high gradient magnetic separation (WHGMS) testing at various field strengths. No practical upgrade in rare earths or gangue rejection was seen in the limited testing.

Flotation

Sighter testwork was undertaken on the HG composite using a fatty acid collector (typical for bastnaesite hosted rare earth feeds) with a sodium silicate dispersant at elevated pH levels. As with the magnetic separation, a practical upgrade in rare earths or gangue rejection was not evident in the limited testing. A standard rougher and cleaner arrangement was tested as shown in Figure 6.3.

A modest (but promising) doubling of the rare earth grade was seen in the combined (1 to 4) cleaner concentrates at ~50% recovery. TREO grades, recoveries and mass distributions are summarised in Table 6.3.

Figure 6.3 Laboratory flotation testwork regime

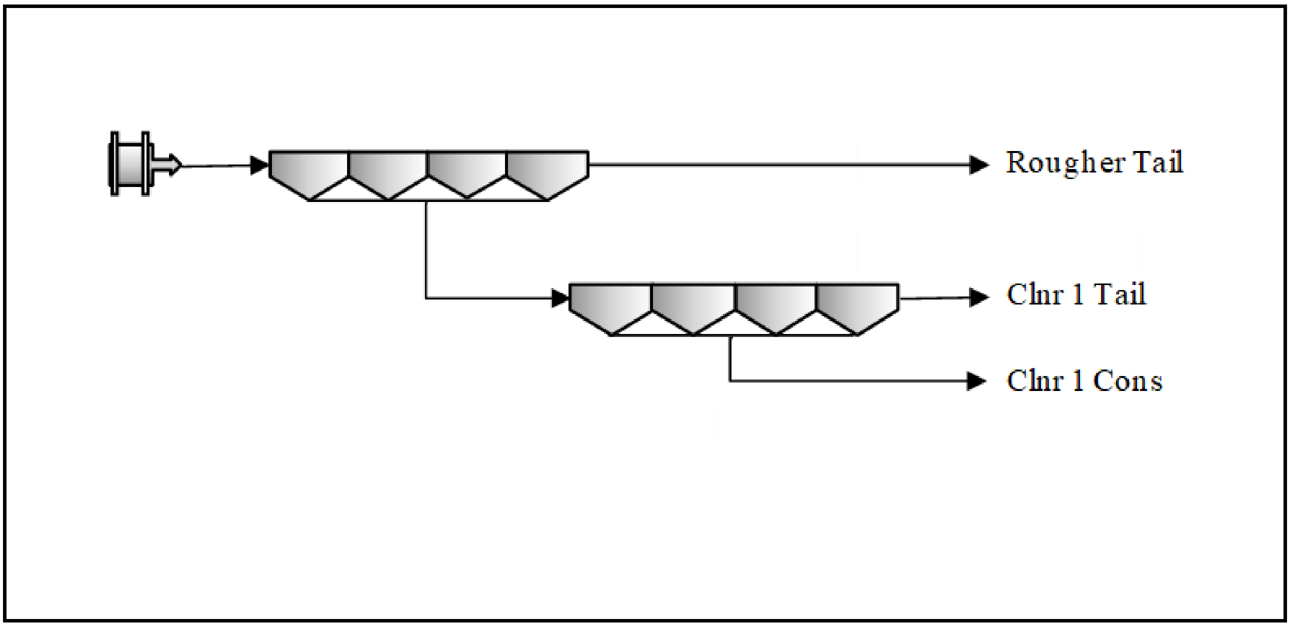


Table 6.3 Sighter flotation results on HG composite

Product	Mass %	TREO by stream		TREO cumulative	
		%	% dist	%	% dist
Cleaner Con 1	7.0	6.72	11.8	6.72	11.8
Cleaner Con 2	12.0	7.62	23.0	7.29	34.9
Cleaner Con 3	5.8	6.43	9.3	7.09	44.2
Cleaner Con 4	2.6	5.83	3.9	6.97	48.1
Cleaner Tail	17.8	3.81	17.0	3.81	65.1
Rougher Tail	54.8	2.53	34.9	2.53	100.0
Calculated head	**100.0**	**3.98**	**100.0**	**3.98**	
Assay head		**3.22**		**3.22**	

Competent Person comments

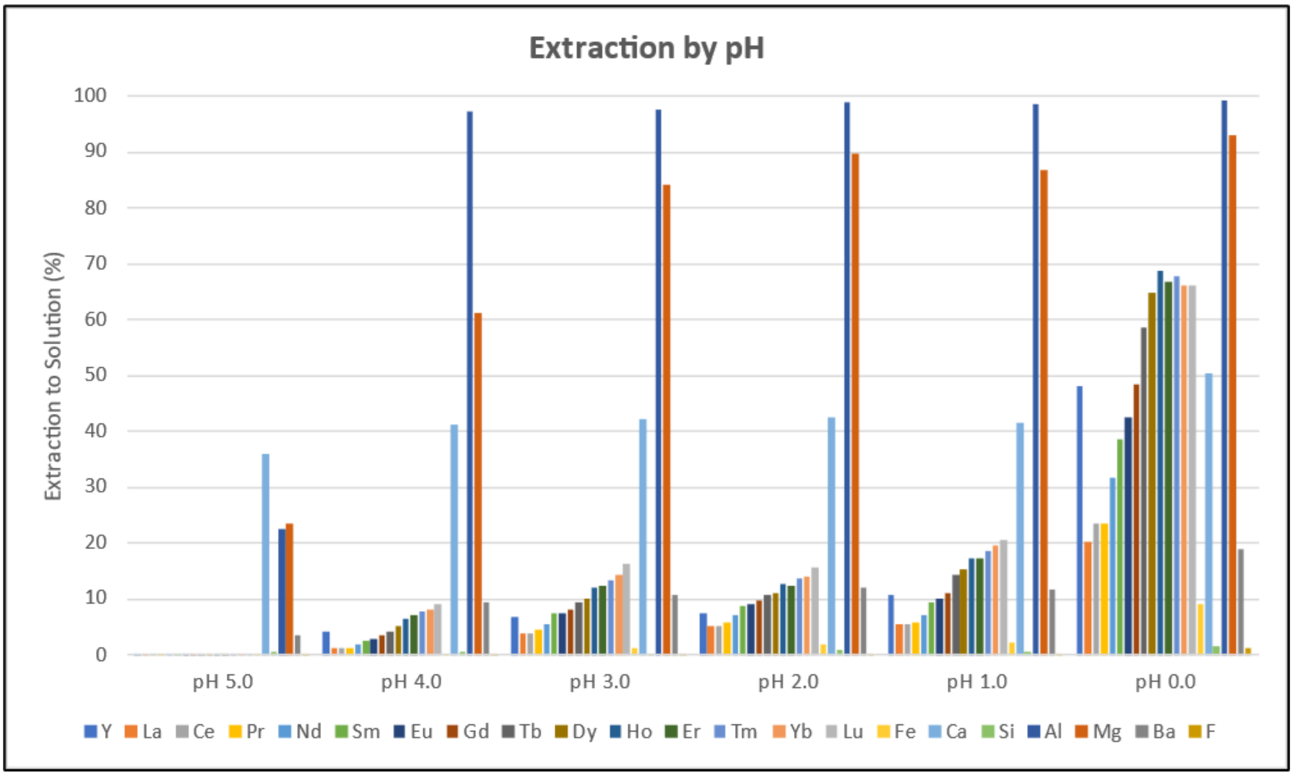
Whilst the magnetic separation didn't show immediate assurance, the flotation produced a doubling of the feed grade without any optimising of feed grind size or reagent regime and conditions. At the time of writing, qualitative mineralogy (QEMSCAN) is underway which will guide further beneficiation programs, qualitative mineralogy (QEMSCAN) was underway which will guide further beneficiation programs.

6.1.3 Hydrometallurgy testwork

Leaching

Mineralogy undertaken on the composite samples indicates significant quantities of calcite (up to 28% by mass) is present. A diagnostic leach was undertaken on a sample of the HG flotation concentrate in weak hydrochloric acid to ascertain if a gangue leach would be appropriate. It was found that more than 40% of the calcium (almost certainly present as calcite), 97% of the aluminium and 61% of the magnesium were solublised at a modest pH of 4.0. Importantly, less than 2% of the high value rare earth contributors (Nd and Pr) were solublised at this pH. The results are presented graphically in Figure 6.4.

Figure 6.4 Diagnostic hydrochloric acid leach on high grade composite



Competent Person comments

The sighter leach testwork undertaken was limited to the gangue leach stage only. It is surmised that a hard leach (strong KCl solution at elevated temperatures) on the gangue leach residue should solubilise the contained rare earths for subsequent recovery. It is noted that the presence of fluorine may inhibit this, however by forming insoluble rare earth fluorides. Should this mechanism result in unacceptable low rare earth extractions, then a cracking process (such as caustic conversion) may be required.

6.2 Process description

6.2.1 Introduction

Rare earth projects generally follow a three-stage metallurgical process of:

- Beneficiation – crushing and grinding (comminution) of the run of mine (ROM) ore followed by physical techniques to separate and upgrade the rare earth host minerals by rejecting the gangue minerals. Techniques used include gravity separation, magnetic separation and froth flotation.
- Hydrometallurgical recovery – comprising of chemical dissolution of the rare earth host minerals using acidic or alkaline processing steps often at elevated temperatures. This is followed by purification steps in order to remove the unwanted elements that dissolved along with the rare earths during the dissolution step. The resultant solution can either be fed directly to a separation stage if this is located on site or precipitated to give a high purity, mixed rare earth chemical concentrate

(typically a Mixed Rare Earth Carbonate or MREC) that will be transported to a remotely located separation facility.

- Separation – the separation of the individual or groups of rare earths into saleable products as determined by the particular end users. Solvent extraction (SX) is almost exclusively used for this purpose.

The first two stages tend to vary significantly across different projects, largely due to the highly variable mineralogy of REE deposits. It is thus essential that appropriate time and resources be devoted to understanding the mineralogy to guide the development of the beneficiation and hydrometallurgy phases. The scope of this study encompasses the beneficiation and hydrometallurgical stages.

6.2.2 Process overview

The possible process design is based on mineralogy, limited testwork and references to other REE projects in operation or at advanced stages of engineering. Altona is targeting a high grade MREC product that is suitable as a feed source to existing third party separation plants. The key design parameters for the processing plant are summarised in Table 6.4.

Table 6.4 Process design parameters

Description	Unit	Value
ROM feed rate	Dry t/a	750,000
TREO head grade	%	2.39
Contained TREO in ROM feed	t/a	17,925
Concentrate mass pull	% of ROM feed	14.3
Mineral concentrate TREO grade	%	10
Recovery of TREO to mineral concentrate	%	60
Contained TREO in mineral concentrate	t/a	10,755
Mineral concentrate feed rate to hydrometallurgy plant	Dry t/a	107,550
Recovery of TREO to MREC	%	80
Overall recovery of TREO from ROM to MREC	%	48
Contained TREO in MREC	t/a	8,604

Note: TREO – Total Rare Element Oxide, ROM – Run of mine, MREC – Mixed Rare Earth Carbonate

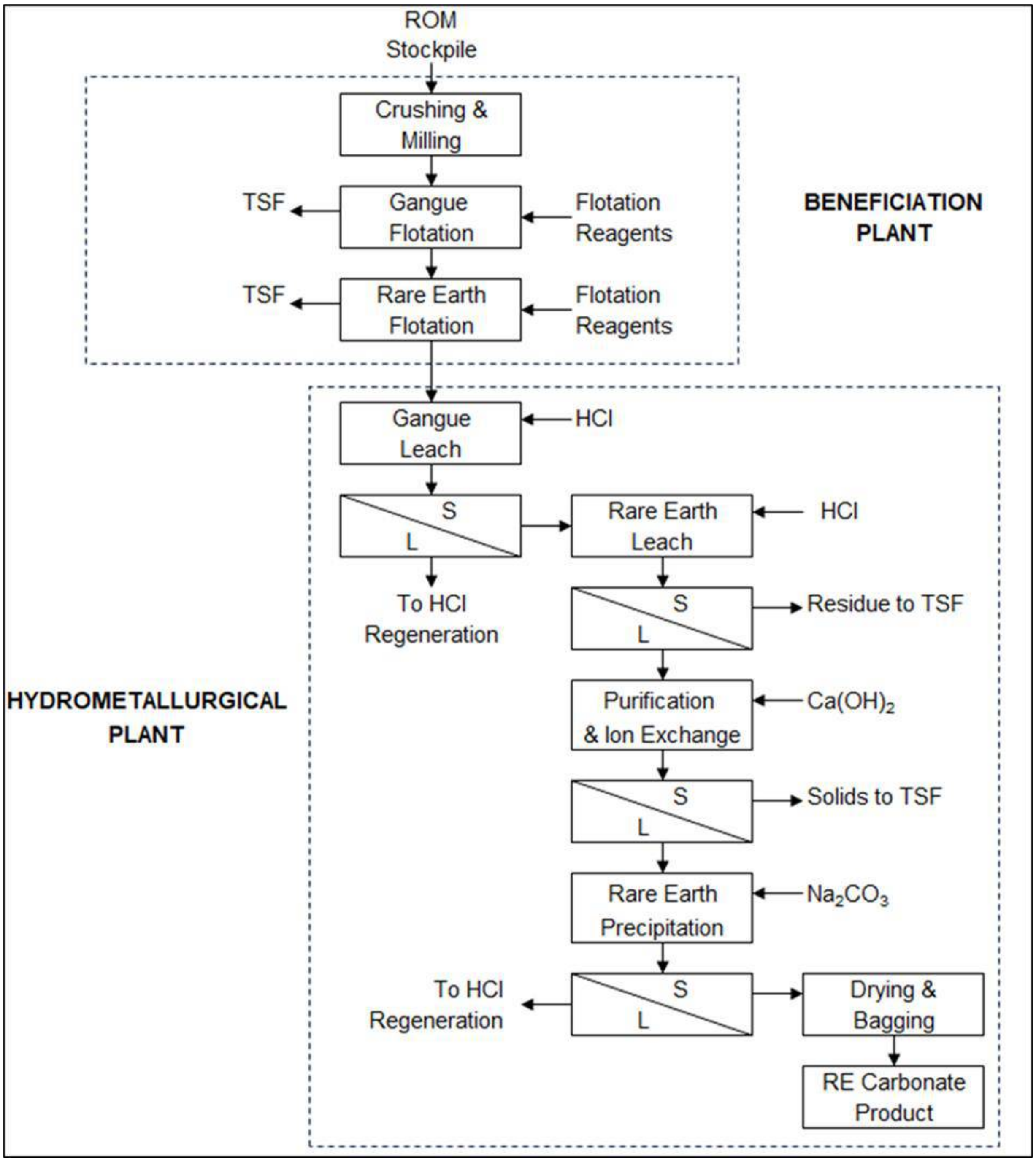
The process flow sheet as illustrated in Figure 6.5 consists of:

- The beneficiation plant which includes for the physical concentration of the rare earth host minerals and the rejection of gangue minerals.
- The hydrometallurgical plant which includes for the chemical recovery, purification and concentration of the rare earths employing a simple hydrochloric acid leach process.

The beneficiation plant comprises the comminution and flotation circuits. The purpose of the comminution circuit is to reduce the size of solid ore particles and thus increase the surface area of solids to enable the liberation of valuable materials that are locked within the gangue minerals. This is achieved by means of crushing and milling.

Flotation is used to upgrade the mineralized material. Flotation is a method of separation, which uses the differing surface properties of the various minerals in the ore. It involves the selective attachment of mineral particles to air bubbles generated in the flotation cell which float to the surface of the slurry and then flow over the lip of the cells into the launders.

Figure 6.5 Process flow sheet



6.2.3 Beneficiation process

Ore receiving and crushing plant

Ore will be recovered from the ROM pad and fed to a ROM bin fitted with an inclined static grizzly to prevent oversize material entering the ROM bin. The bin discharges through a feed chute onto a vibrating grizzly feeder. Undersize material discharges via a chute onto the secondary crusher feed conveyor. Oversize material discharges into the primary jaw crusher. Jaw crusher product, discharges and combines with the vibrating grizzly feeder undersize.

The secondary crusher feed conveyor discharges onto the double deck crusher feed screen. The screen (bottom deck) undersize material discharges onto the screen undersize conveyor. The material is transferred through a chute onto the mill feed silo. The top deck screen oversize discharges onto the secondary crusher feed conveyor and the bottom deck screen oversize discharges onto the tertiary crusher feed conveyor.

The secondary crusher is a standard head cone crusher with the product is recycled to the double deck screen. The tertiary crusher is a short head cone crusher with the product also recycled to the double deck screen.

Mill feed silo

The mill feed silo has a nominal capacity of ~2,000 t, equivalent to approximately 24 hours of milling capacity. Crushed ore is drawn from this silo using withdrawal vibrating feeders. Each feeder discharges onto the ball mill feed conveyor. Ball mill feed rate control is achieved using the variable speed vibrating feeders.

Milling

The ball mill operates in a closed circuit with a cyclone cluster to produce a product size P_{80} of approximately 53 µm in the cyclone overflow which is envisaged to provide the necessary degree of liberation for effective flotation. The cyclone overflow will gravitate into the gangue flotation feed tank.

Flotation

The flotation process consists of two distinct stages: gangue flotation and rare earth flotation.

The gangue flotation is designed to remove the majority of the calcium bearing gangue minerals such as calcite, fluorite and ankerite. These typically require a weak collector such as a fatty acid and is undertaken at ambient temperatures. As this is a reverse flotation process, the gangue reports to the flotation concentrate which is subsequently cleaned in a number of stages to recover the rare earth hosted bastnaesite minerals. This concentrate is thickened before pumping to the TSF.

The tailings from the gangue flotation contains the rare earth hosting bastnaesite mineral as well as non-calcium gangue minerals (silicates, feldspars etc). This stream is re-ground to further liberate the bastnaesite minerals and/or polish the mineral surfaces of the gangue flotation collector.

The rare earth flotation typically uses a stronger collector (fatty acid or hydroxamate) at elevated temperatures (40° to 50°C). Multiple roughing and cleaning stages are typically employed. The tailings from this circuit are sent to the gangue flotation thickener. The concentrate is sent to the rare earth mineral concentrate thickener.

Plant utilities (water)

The water reticulation system is designed to provide the following water services:

- Raw water.
- Filtered water.
- Fire water.
- Potable water.
- Process water.

Raw water is pumped from source into the raw water tank. The raw water tank overflows into the process water tank for make-up water. In addition, the raw water is pumped through the sand filter plant into the filtered water tank. Filtered water is distributed to reagents make-up, gland services, potable water treatment plant and fire systems. Process water is stored in a process water tank and is distributed to the plant by process water pumps.

Flotation reagents

Flotation reagent storage, make-up and distribution systems are typically required for the following:

- Gangue flotation collector.
- Rare earth flotation collector.
- Activator (typically sodium silicate).
- pH modifier (sodium hydroxide or sodium carbonate).
- Depressants.
- Dispersants.

6.2.4 Hydrometallurgy process

Calcite leach

The thickened rare earth mineral concentrate slurry is pumped to the first agitated fibreglass reinforced plastic (FRP) tank thereafter the slurry cascades to agitated FRP tanks where calcite leaching occurs using a weak HCl solution maintained at pH 4. The purpose of the calcite leach is to remove/solubilise the calcium contained within the REE host minerals of bastnaesite as well as residual calcite not rejected by the flotation process.

The leach slurry is thickened and filtered. The solution (thickener overflow and filtrate) is pumped to the HCl regeneration circuit with a small amount recycled back to the first leach tank to adjust the pulp density and build up the calcium tenor to improve HCl regeneration efficiency. The calcite leach residue from the filter is repulped with rare earth leach thickener overflow before being pumped to the rare earth leach section.

Rare earth leach

The purpose of the rare earth leach is to solubilise the rare earths by dissolving the bastnaesite host mineral. Repulped slurry from the calcite leach circuit is pumped into a series of cascading agitated FRP tanks and leaching is done with a strong 20 % (v/v) HCl solution at approximately 80ºC. The rare earth leach slurry is pumped to the rare earth thickener. The thickener underflow is sent to the leach filter which combines the filtrate with the thickener overflow and is pumped to the purification circuit.

The filter cake contains the depleted leach residue and is repulped and pumped to the waste treatment tank.

Purification

The purpose of the purification section is to precipitate solubilised gangue elements as their hydroxides. Slake lime (calcium hydroxide) is added to a series of FRP tanks to precipitate iron, aluminium and thorium. The resulting slurry is then thickened and filtered before the filter cake is repulped and pumped to the waste treatment tank.

The thickener overflow and filtrate is now essentially a rare earth chloride solution with some uranium contamination. This solution is treated via standard ion exchange (IX) columns to remove the uranium. The resulting purified rare earth liquor is sent to the MREC precipitation circuit.

Mixed rare earth carbonate (MREC) precipitation

The purified mixed rare earth liquor is subjected to a final precipitation step whereby a sodium carbonate (soda ash) slurry is added in a series of cascading, agitated FRP tanks.

The slurry from the agitated FRP tanks is fed to the final rare earth precipitation thickener. The thickener overflow and final MREC filtrate is recycled to the raw rare earth precipitation circuit and/or HCl regeneration. The thickener underflow is filtered then the rare earth filter cake is dried, cooled and bagged for shipping.

Hydrochloric acid regeneration

The HCl regeneration circuits involves contacting calcium chloride ($CaCl_2$) solution with sulphuric acid (H_2SO_4) to produce HCl and insoluble gypsum ($CaSO_4$). This is a well-documented process that enables regeneration of HCl on remote sites using H_2SO_4 that is also generated on site.

The calcite thickener overflow and filtrate solution together with the filtrate after the MREC precipitation is pumped to the HCl regeneration holding tank. The $CaCl_2$ solution is pumped to a series of cascading agitated FRP tanks in which 98 % H_2SO_4 solution is added. $CaCl_2$ can also be supplemented if required. The general form of the equation is:

$$CaCl_{2(aq)} + H_2SO_{4(l)} \rightarrow CaSO_{4(s)} + 2HCl_{(aq)}$$

The HCl and gypsum slurry is pumped to a thickener and filter whereby the thickener overflow and filtrate comprises of the regenerated HCl which is sent to a holding tank for plant distribution.

The gypsum filter cake can either be dry stacked or repulped and pumped to the TSF.

Sulphuric acid production

A commercial sulphuric acid burner plant is fed solid sulphur prill to produce the H_2SO_4 required for the HCl recycle process. There is an option to add a steam turbine power co-generation package to the package if excess steam is available. For the purpose of this study, it is assumed that the steam produced will be used only for heating the flotation and leaching circuits.

Reagents

Hydrometallurgy reagent storage, make-up and distribution systems are typically required for the following:

- Sulphur prill.
- Calcium chloride flake.
- Slaked lime.
- Sodium carbonate.

6.3 Ongoing testwork

Further detailed metallurgical studies for Monte Muambe are underway and currently focused on advanced metallurgical testwork. A 70 kg representative ore sample is with Auralia Metallurgy in Perth (Australia), and another 100kg ore sample has been received by SGS Lakefields in Canada. The sample at SGS Lakefields will first undergo extensive feed characterisation including Electron Microprobe Analysis and TIMA-X analysis. TIMA-X is designed to provide quantitative mineral speciation and distribution, as well as characterisation, grain size attributes, degree of liberation and associations of minerals of interest.

Following feed characterisation, test work will focus on producing a high-grade Rare Earth concentrate in order to improve the economics of the Mixed Rare Earth Carbonate production process. The possible separation and recovery of fluorspar, another critical raw material present in the ore at Monte Muambe, will also be assessed. Flotation test results are expected in Q2 2024.

6.4 Competent Person comments on metallurgy and processing

When referring here to a flowsheet, it does not imply the study flowsheet will be necessarily retained, as alternative processing routes could be selected as the sighter testwork progresses and associated technical and cost-benefit assessments are implemented.

The selected flowsheet is deemed reasonable based on comparing the mineralogy with operating plants of other rare earth deposits in advanced stages of development (i.e. have undertaken pilot plant verification).

7 PROJECT INFRASTRUCTURE

7.1 Site layout

All required infrastructure, the accommodation camp, process buildings, stockpiles, water resources, and TSF are located within the current Project's Licence boundaries. Site infrastructure is required both inside the crater and outside to service these facilities.

A conceptual site block plan (SBP) locates the main accommodation camp outside of the crater to minimize dust, noise and radiation exposure, whereas the process plant, mining contractors' workshops, TSF and associated infrastructure are deployed inside the crater, arranged to minimize the physical footprint and in close proximity to the two main pits.

The planned mine infrastructure and process areas are, as far as practical, consolidated to reduce materials handling distances (including that of ROM ore) and laid out to exploit or adapt to the topography as well as accommodating future expansion of selected process units and exploitation of new pits.

7.2 Power

The electricity supply is managed by Electricidade de Moçambique (EDM). The largest power generation plant in the country, which is still responsible for the bulk of installed capacity, is the Cahora Bassa hydro dam, operated by the government owned Hidroelectrica de Cahora Bassa (HCB). HCB sells 65% of its existing generation to South Africa, and the remaining 35% is sold to the northern regions of Mozambique and to Zimbabwe. HCB's operations are located on the Zambezi River in Tete Province (International Trade Administration, 2021).

The government also has a number of projects to both increase capacity as well as transmission, large projects notably include:

- The Cahora Bassa North Bank Hydropower expansion, to add an additional 850 to 1,300 MW.
- The 1,500 MW Mphanda Nkuwa Hydropower (60 km downstream from Cahora Bassa dam and 70 km upstream from Tete city).
- Gas-to-power program: Thermal power plants centred on the Temane-Pande gas fields in central Mozambique and the Rovuma basin gas fields in northern Mozambique. These projects both include accompanying transmission line projects in line with the Mozambique government's MEFA Programme (Mozambique Energy for All) which aims (i) to increase stability of the Mozambican power system, (ii) enable a large expansion of power sales to Southern Africa Power Pool (SAPP), (iii) expand access to electricity and (iv) assist in improving EDM's financial sustainability.

The first Independent Power Projects (IPPs) in Mozambique came online in 2015. The Gigawatt 120 MW gas-fired power station at Ressano Garcia plant was commissioned in 2015 under a Power Purchase Agreement (PPA) with EDM.

Gas-based generation in Mozambique is expected to increase by 18.1% annually through 2025. Mozambique's first utility-scale solar power plant, a photovoltaic plant with a capacity of 40 MW, was commissioned in Zambézia Province in 2017. Additionally, there are numerous other greenfield opportunities for both solar and wind projects. Mozambique has set significant targets for the development of its electricity sector, including at least 2,300 MW of new installed capacity by 2030 and about 5 million new connections, both on grid and off grid, to achieve universal access to electricity by 2030 (AFDB, 2021).

7.3 Project power

The Company is 36 km from the Moatize mine transformer yard. A grid overhead power line (220 kv) is within 4.5 km of the Project. In addition, there is a medium voltage line getting to Mualadzi (24 km north of Monte Muambe), from which power can be brought to Monte Muambe.

Diesel generators will be used during construction and commissioning, and as backup for infrequent events of grid failure.

The 18 MW electrical power maximum demand of the Monte Muambe site will be provided at 11 kV, 50 Hz by a hybrid power generation plant. The plant will comprise a backup diesel-powered electrical generator station and solar photovoltaic (PV) power station supported by a battery energy storage system. Cables will link the power station substation to the process plant substation.

Solar PV generation is expected to contribute approximately 25% to the overall generation. The solar PV power station will be contained in a separate area, 800 m upwind from the process plant.

The backup diesel generators will be located adjacent to the process plant facilities and be capable of providing the maximum power demand with a N+1 level of redundancy for maintenance shutdowns or failure of a single generator. All diesel generator units are planned to be individually housed in acoustic containers and located in a section of the beneficiation plant that can be bermed off to further isolate noise from the general plant site. Fuel storage and a control room will be included in the installation.

Site electrical power will be distributed by means of overhead lines for remote facilities, such as the borehole field, accommodation camp, tailings storage facility, and explosives magazine. The process plant, being in proximity to the power station, will have electrical power distributed by cables buried or on racks.

7.3.1 Power requirements and costs

A total of 5 x 2 MVA diesel generators will be required for the initial phase to provide power during construction and in the interim till the EDM bulk power supply is available. This is an estimate of the initial power requirements before the planned mine comes into full production and will be in use for approximately 12 months depending on the construction program. The generators can then be phased into the initial phase of production then provide standby power of 8 MW when the bulk power supply is commissioned.

Table 7.1 lists the power supply requirements and costs to meet the power demand of the planned mine during the construction and initial phase. The assumption is that the diesel generators have 12 hours run time at 75% load with a tank capacity 4,000 litres. The diesel generator costs are rounded off and exclude switchgear and distribution.

Table 7.1 Power supply requirements and costs

Power generation	Total continuous power supply (kVA)	Capex cost (35% accuracy)	Opex cost (35% accuracy) At 75% load 325l/hour per 2 MVA generator – Diesel cost @ $1.53 per litre	Opex cost (35% accuracy) At 75% load 5 x 2 MVA generators
Phase 1- 5 off 2 MVA diesel generators construction phase and initial start-up -	10 000	$1,500 000 per unit = $7, 500 000	$497.25 / Hour = $11 934 / 24 hr day = $358 020 / Month	$358 000 / Month/ 2 MVA Generator = $1 790 100 per month for 5 2 MVA generators

The power demand is estimated at 18 MW, this is summarised in Table 7.2. The data is based on the information received from Altona management as there is no mechanical equipment list available for this project to determine the total installed power at this stage.

The ball mill will be the largest power user and thus the start-up energy will be significant. It is assumed the ball mill will be driven by a slip ring wound asynchronous motor which will be equipped with a liquid rotor starter (LRS) which is more cost effective than a variable speed drive (VSD).

The power plant will be adequately equipped to supply sufficient power during start-up without interruption to the rest of the plant equipment. This will have enough capacity to mitigate any significant voltage drop that may lead to interruptions in the power supply to the plant equipment.

Table 7.2 Plant and infrastructure power requirements

Project load	Mine in construction phase	Mine in full production continuous power demand (kW)	Maximum start-up demand (kW)
Process plant	3,500	14,000	

Project load	Mine in construction phase	Mine in full production continuous power demand (kW)	Maximum start-up demand (kW)
Off-site infrastructure	500	1,000	
Construction	1,000	500	
Ball mill	0	2,500	4,000
Total		**18,000**	

7.4 Water

Management consider that a bore well-field will be the most optimal solution for the Project's water supply. Groundwater generally has a high total dissolved solids (TDS) load (about 1,000 mg/l) and this will require water treatment prior to use in the plant and as potable water. Currently borehole water is abstracted from a converted exploration borehole located in the central part of the crater.

The closest point to the Zambezi river is approximately 30 km from the Project site. This may be considered as a possible abstraction source in the future.

The water demand for the Monte Muambe site will be supplied by on-ground overland pipelines from the bore field. Various sites for bore water have been identified and well tested. The bore field selected will provide water to the accommodation camp, process plant and camp. Dewatering from the open pits is also expected to increasingly contribute to the plant's water supply during the LOM.

Three qualities of water will be required on the Monte Muambe site.

- Raw water for bulk use in the process plant and mining for dust suppression; this will require an earth-lined dam to act as a buffer storage facility.
- Potable water for eye wash stations, food preparation and human consumption.
- Demineralised water for steam production.

A containerised potable water treatment plant comprising activated filter media filtration, softening, chlorine dosing and UV disinfection will be used to treat the water required for potable water services. A containerised demineralisation plant will be used to treat the raw water required for the boiler plant. Fire water distribution and potable water distribution will be included in the overall Project design.

The retentate from the potable water plant and the blowdown from the demineralisation plant will be discharged along with the waste process streams to the TSF. The supernatant water from the TSF will be reclaimed and recycled to the process plant with zero discharge. All wastewater arising from domestic use including from kitchen, canteen, and laundry will form part of sewage. Wastewater from the sewage treatment will be directed to either a spray field during the dry season or discharged through a leach field during wet season.

A TSF water return system will be considered in the TSF design.

7.4.1 Water distribution

Water will be pumped by HDPE pipeline incorporating valve chambers, thrust blocks and protective earth mound from the borehole wellfield to an elevated site on the crater rim into a dual compartment Braithwaite type steel panel tank founded on a reinforced concrete base and 1-m tall plinths servicing the following areas :

- Inside the crater:
  - Contractor laydown area
  - Office complex
  - Infrastructure and vehicle workshops area
  - Stores area
  - Mining infrastructure area
  - Container yard
  - Beneficiation plant

  - Weighbridge.
- Outside the crater:
  - Diesel tank farm
  - Sports fields
  - Kitchen, dining and recreation buildings
  - Change house, laundry and boiler room
  - Accommodation for C, D and E-class personnel.

7.4.2 Water demand forecast

This will be defined during PFS testwork and design, with an accuracy of litres/ day. Based on this planned demand the potable water reservoir will be made large enough for a two-day storage capacity.

7.4.3 Reticulation

Potable water from the Braithwaite-type reservoir is distributed by gravity feed in a 140 mm diameter HDPE pipeline configured as a ring-feed. A series of gate valves are introduced to isolate supply areas for maintenance and scour valve chambers are positioned at strategic points to drain pipelines, as maintenance is required. This ensures a reliable supply of potable water to all consumers.

The potable water pipeline is bedded in earthen berms which run on grade. At road crossings, the pipeline is concrete encased with 15 MPa mass concrete to prevent damage.

7.4.4 Raw water dam / process water dam

A HDPE-lined raw water dam is envisaged with an effective storage capacity of 5,000 m^3 with a base footprint of 30 m x 30 m with a depth of 3.5 m and side slopes of 1V:2.5H. Raw water supply from the wellfield, operating at 55 ℓ/s, can fill the dam in approximately 24 hours.

A 2 mm HDPE smooth flexible geomembrane is envisaged for the lining system. This infrastructure is constructed during the early works phase of the project as the specialist HDPE lining supplier requires the earthworks for the dam to be constructed and for attendance by the main contractor to supply labour resources and plant to assist with placement of the liner. A reinforced concrete spillway and chute is provided in the dam wall to prevent overtopping due to mechanical failures or major storm events.

A water bowser filling station is provided adjacent to the dam for filling water trucks for dust suppression, accessible by 6 m wide road constructed off the primary access road and with a loop route back to the beneficiation plant for access to the raw water dam for firefighting.

A 250 mm diameter suction pipe, surface mounted to the earth embankment dam and anchored to the base of the dam, is provided for abstraction of water into the pump station, which is delivered to the plant or to a containerised demineralisation facility to process the raw water prior to use in the boiler plant.

During periods of low demand, raw water can be pumped out of the crater for consumption by the local community for agricultural purposes.

To mitigate loss of dam capacity from the settling of particles in suspension, a reinforced concrete silt trap is incorporated at the raw water dam inlet. Sluice gates control the flow of raw water into the two compartments. Using the sluice gates, one compartment can be isolated, allowing removal of excess silt deposition and for maintenance without disrupting water supply to and from the dam.

7.4.5 Waste water

Two treatment works are envisaged, one for the accommodation camp and facilities outside of the crater, and another to service the plant requirements and facilities in the crater. Outlying areas such as the gatehouse, explosives bunker, wellfield, etc. will be serviced by environmental toilets and a small potable water tank on an elevated stand.

All wastewater arising from domestic use including from kitchen, canteen, and laundry is treated as sewage. Generally, 80 % of potable water consumption is returned as sewage discharge. Based on the potable water demand the effluent production per day can be planned.

All sewerage effluent is reticulated to containerized waste water treatment modules. The containers are placed on ground slabs. This system will be a design and supply package by specialists. It is proposed that multiple modules treating 70 m^3/day effluent by submerged aerated fixed film (SAFF) method satisfy the requirements for water quality standards of South African (Section 39 of the Water Act of 1998 – Act 36 of 1998), and are deemed appropriate for this facility in Mozambique.

Modular plant units can be rented to expand capacity as manpower numbers increase (and reduce). A 400 PE unit catering for 70 litres/ person/ day requires a 36-hour buffer/ retention tank and caters for a maximum of 350 people, operating 24-hours per day. A buffer/retention tank is not included in the modular package, which requires a separate civil structure to be constructed. A 3-compartment structure 6.0 m long x 2.5 m wide and approximately 2.5 m deep provides 36 m^3 buffer capacity with 500 mm freeboard. The number of buffer tanks depends upon the number of wastewater treatment modules deployed.

The plant will be charged by a gravity sewer pipeline unless the topography required the effluent to be pumped. Details of pumping the effluent into the treatment plant are yet to be designed as well as access for maintenance and annual removal of sludge by means of pumping into tankers for disposal at a suitable hazardous waste site or drying bed facility.

Various sundry equipment and infrastructure required for the waste water treatment plant include grease traps, discharge pipe and pump system to dispose of treated effluent, standby equipment, analytical instrumentation and reagents for testing, including storage facilities.

Commissioning of a modular plant requires approximately 2-months for micro-organisms to achieve full operational specifications. If this standard is met, water may be discharged as surface runoff, or used for other purposes, such as agriculture. Water quality and the treatment system must be routinely checked in order to ensure compliance. During the commissioning of the plant and during maintenance of the plant, effluent will have to be stored in a buffer tank (larger than that required for normal operation of the plant) or bypassed into the dirty water dam to prevent contamination to the surrounding environment.

Wastewater from sewage treatment is directed to either a spray field during the dry season or discharged through a leach field during wet season. Sludge will be disposed of in the waste dump.

7.4.6 Storm water management

Storm water infrastructure caters for natural overland flow that requires to be directed around man-made terraces, dumps and stockpiles, the pit and across roads, to continue overland into natural drainage gullies, as well as the conveyance and collection of rainfall runoff, decant water and spillage off terraces, TSF, stockpiles and waste dumps that may not be discharged back into the natural environment.

Storm water is considered clean water (uncontaminated) when surface runoff is remote from the plant, roads and terraces for workshops, waste dumps, ore stockpiles and areas where ore is handled. Clean water is dispersed back into the environment to recharge groundwater or to drain away in natural gullies.

Rainfall runoff from ore stockpiles and soil dumps that originates down-slope of clean water cut-off channels, is deemed to be contaminated and is collected in dirty water retention ponds. Contaminated spillage from the plant terraces, workshops and stores, etc., is directed to lined channels which discharge off the terraces in chutes. Where required, stilling basins or silt traps retain solids before the dirty water discharges into evaporation ponds or dirty water dams.

Dirty water runoff and decant water from the tailings storage facility TSF is collected into retention dams and pumped back to the treatment plant. Designs for the management and re-cycling of this water is dealt with in the detailed design of the TSF and excluded from general site infrastructure scope of work. Similarly, mine pit de-watering including direct rainfall within the pit footprint is collected and pumped to the treatment plant and also forms part of the water-mass balance calculations.

All drains are designed to convey the 1:50 year return period flow without overtopping. A minimum slope of 1:100 ensures that the drainage channels are self-cleaning. Where the topography is steep, drop structures will be introduced to reduce storm water velocity.

Road storm water drainage

Storm water run-off from roads that are not in areas classified as contaminated zones, is conveyed in gravel-lined mitre drains alongside road edges and directed to the surrounding environment where the runoff is classified and clean. This includes roads outside the crater in the accommodation camp and the 30 km upgraded mine access road.

Drains are generally unlined earth ditches but concrete-lined mitre drains are used where velocities would scour unlined drains or where materials handling equipment would damage un-lined drains. Where storm water is diverted under roadways, culverts are utilized.

Roads within the tenement area, including the entrance gate, mine offices, mine camp and accommodation facilities, container yard, mine electrical sub-station and workshops and stores, or any other man-made terrace or road, are deemed to be in the contaminated zone and runoff is collected in concrete lined channels and conveyed into the dirty water system.

7.4.7 Drainage channels

Two types of drainage channels will be used, namely, clean and dirty water drains:

- Clean water drains cut off and convey surface storm water around the plant and mining areas to protect and prevent flooding. This storm water is discharged into the surrounding environment (in accordance with local environmental legislation), eventually running out of the crater.
- Dirty water drains convey contaminated water within the plant and mining areas that is not suitable for discharge into the surrounding environment. This includes process plant, stockpile, waste dump, stores, workshop and wash-bay run-off.

No mixing of contaminated and non-contaminated water is permitted. Sites for centralized dirty water dams will be suited to the topography and to facilitate gravity conveyance from localized dirty water retention ponds. Separation of solids from the contaminated water in silt traps allows for removal to appropriate hazardous waste sites, in either solid or sludge form.

Earth and concrete-lined storm water drains generally comply with the criteria summarised in Table 7.3 and Table 7.4.

Table 7.3 Earth lined cut-off drains

Type	Unit	Type 1: Earth cut-off drain
Top width	mm	2,500
Base width		500
Depth		500
Side slope		1:2
Slope		Varies (min 1:100)
Q (minimum slope)	m^3/s	2.110
Velocity at minimum slope	m/s	2.813
Total length required	m	TBC

Table 7.4 Concrete lined channels

Type	Unit	Type 1: Concrete channel	Type 2: Concrete channel	Type 3 Concrete channel	Type 4 Concrete channel
Top width	mm	1,000	1,400	800	600
Depth		250	350	200	150
Side slope		1:2	1:2	1:2	1:2
Slope (varies) Longitudinal		min 1:100	min 1:100	min 1:100	min 1:100
Q .@ min slope	m^3/s	0.193	0.474	0.107	0.05
Velocity @ min slope	m/s	1.55	1.94	1.33	1.101

7.5 Landfill site

A general landfill site of approximately 14,500 m^2 (145 x 100 m) is envisaged. The earthworks terrace is shaped into a dish structure underlain by a geomembrane liner with an under-drain layer barrier protection complying with Class C landfill criteria (subject to confirmation of hazardous waste classification). This comprises of 300 mm thick finger drains of geotextile covered aggregate over a 100 mm silty sand protection layer for a 1.5 mm HDPE geomembrane. This barrier system overlies 2 x 150 mm thick compacted clay layers bedded on a 150 mm compacted in situ base preparation layer.

Leachate is collected in the finger drains and flows into a dirty water evaporation pond. An overflow pipe that is connected to a sewerage line returning to the treatment plant is integrated into the pond to manage excessive runoff.

The facility is to comply with the South African Department of Water Affairs and Forestry (or latest edition) – Minimum Requirements For Waste Disposal By Landfill and Minimum Requirements for the Handling, Classification and Disposal of Hazardous Waste A, reviewed by the EIA consultants for the project, and is deemed appropriate for this facility in Mozambique.

Cut off clean-water drains upstream of the landfill site divert overland storm water runoff flow away from the terrace to prevent contamination. Dirty-water drains divert contaminated surface runoff to the evaporation ponds.

The following waste types are separated and allocated to dedicated areas at a salvage yard for recycling and disposal off site, or incinerated on site:

- Recyclable waste:
  - Cardboard, paper, plastic, wires, tyres
  - mechanical parts and scrap metal
- Composting waste; food, vegetation.
- Hazardous waste:
  - Laboratory waste
  - Oil and fuels
  - Medical waste
  - Radioactive waste.

7.6 Road and general access

Monte Muambe is approximately 50 km east-south-east of the city of Tete. The capital city of Tete Province is located on the banks of the Zambezi River and at the site of two bridges which cross the Zambezi (one currently being rebuilt following recent flooding). The city of Tete also sits at the critical juncture of the all-weather tar roads linking Harare, Zimbabwe with Blantyre, Malawi, as well as the port cities of Quelimane and Beira in Mozambique. Additionally, Tete is immediately adjacent to and adjoining the coal mining town of Moatize, which has a railway linking it to the port city of Beira. At its closest point the Moatize-Beira railway line is approximately 20 km north-east of the Monte Muambe crater.

The city of Tete is also serviced by international flights to Tete's Chingozi Airport by Airlink out of South Africa, as well as locally by Mozambican carrier LAM – Mozambique Airlines.

Primary access to the site for cargo and personnel is by road. Tete is 134 km from the Project; and the town of Cana-Cana is 67 km from the Project. From Blantyre (Malawi), the Project is located 244 km by road from the south, on the M1 (passing through the towns of Doa and Mecito); and 231 km by road from north on the M6, passing through Cana-Cana. As the crow flies, the Malawi-Mozambique border is 35 km in an easterly direction from the Project boundary; however, the closest official border crossing is at Mwanza-Zobue, approximately 130 km to the northeast, with travel time of more than 3.5 hours.

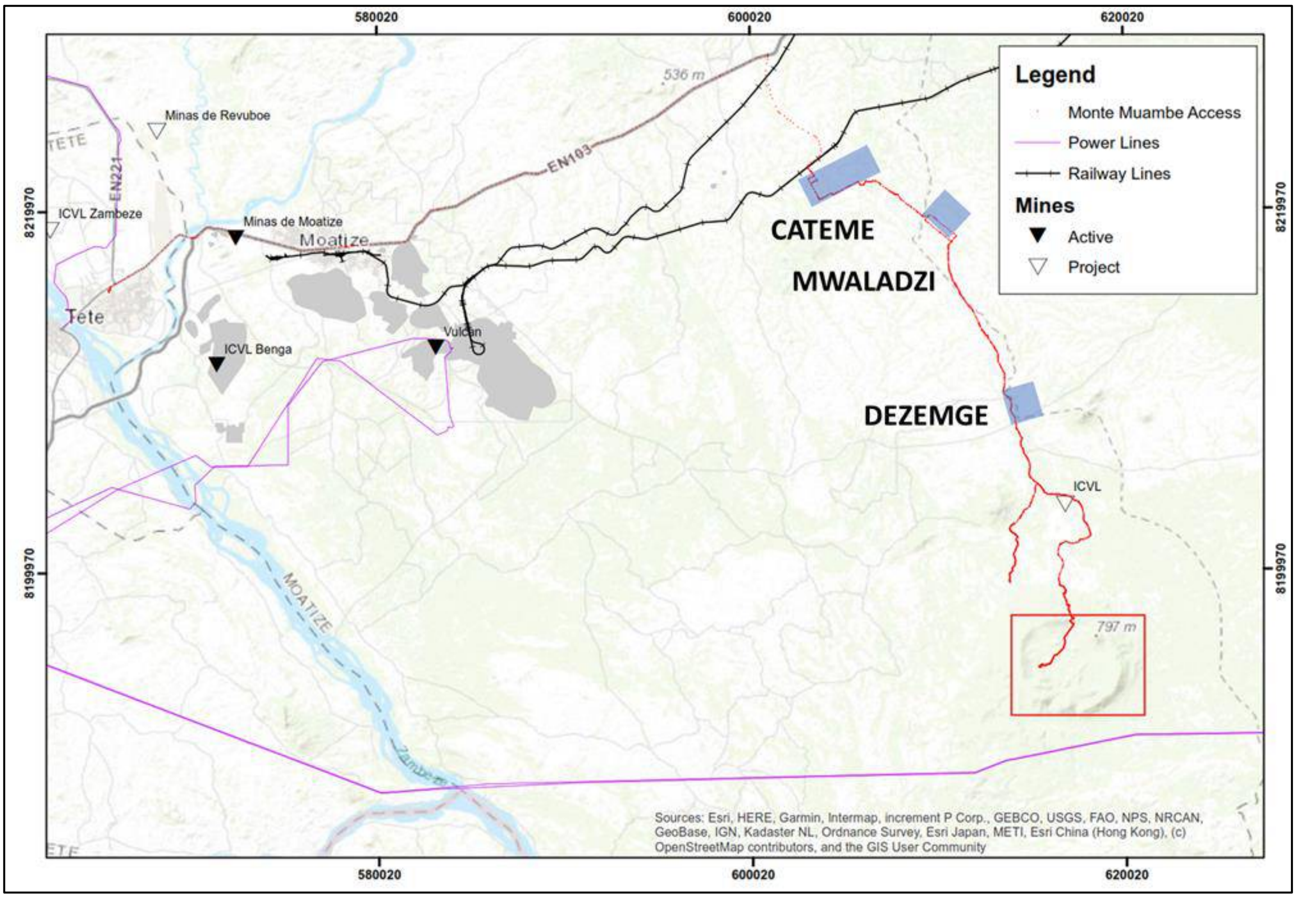
The Project is primarily accessed from the northern side. A tarred, single carriageway (N7) extends from Tete, passed Moatize coal mine in a north easterly direction, a total distance of 70 km. There is a right turn onto a tarred single carriageway extending in a southeasterly direction for 10 km to Cateme. From Cateme, a 35 km gravel road is used to access the Project site; this road passes through the villages of Mwaladzi and Dezemge (Figure 7.1).

The road from Dezemge to the Project site will require upgrading, including by-passes around villages. The road climbing from the foot of the mountain to the existing camp and into the basin will need to be redesigned to ensure a maximum slope of 10%. Estimated cost of local mine road refurbishment only, without bridge strengthening is approximately $200,000.

Inside the basin, where the mine and plant facilities will be located, the topography is gentle. Existing dirt tracks will require widening and upgrading using locally sourced road metal.

For this Project, it is a requirement to upgrade the gravel road between the village of M'cacama and the mine, which includes construction of culvert bridge over the Teixeira River. This work lies outside of the Licence area but is essential infrastructure for the transport of mine product to market and for delivery of consumables to the mine. The cost of the road refurbishment and bridge is noted elsewhere.

Figure 7.1 Location of the Project, licence LPP7573L in Tete Province, Mozambique



Source: Altona, 2023

7.6.1 Mine roads

The main access road into the crater is routed on a north-south axis, passing the temporary construction camp (future Accommodation Village) before traversing down the crater rim to the entrance gate at the plant. Use of the access road is limited to delivery of reagents, equipment and consumables and for transportation of ore product in addition to normal commuting of the labour force. This road is designed for high traffic volume and a range of heavy vehicle axle loads, especially during the construction stage. Gradients and turns to traverse the crater rim require robust construction and good drainage and erosion protection, as well as providing low resistance to skidding. Where appropriate, the road is widened and surfaced, in addition to stabilization. Routine road maintenance and frequent repair work is a requirement, unless the road is surfaced.

Secondary gravel roads provide access for construction and maintenance of infrastructure facilities such as the waste dumps, fuel depot, explosives bunker, waste water treatment works, water storage dams and wellfield. These are deemed to be single lane low traffic volume roads and are designed to the lowest standard that is appropriate.

The mining contractor's roads and pit haul roads are constructed as part of mining operations.

For road construction it is assumed, in the absence of geotechnical information, that the properties of the in-situ soils as the sub-grade layer of the road-bed or terrace can be pre-collapsed by excavating a box-cut to an appropriate depth below the road-bed (or terrace) and dynamically compacting the in-situ horizon using an impact roller. The excavated material is replaced in compacted layers into the box-cut and the structural layer work above natural grade is then built up with suitable quality material from the borrow pits, compacted layers. For unpaved roads and terraces, the selection of materials for pavement design is based on a combination of availability, economic factors and experience of construction in that environment. The recommended design procedure should also adhere to standard material specification defined in national or local authority standards and guidelines.

Allowance is made for typical culverts, bridge and conveyor crossings. Side drains for the roads are suited to typical cross sections for flat, rolling and hilly terrain. Further geotechnical testing and laboratory work is required for detailed design development with the available resources on site. Particularly vulnerable terraces and roads require concrete surfacing such as the container handling yard and other heavily trafficked roads may need to be upgraded and the wearing surface chemically stabilized. This includes an appropriate standard for upgrading the 30 km access road to the mine and to diesel tank farm facility based on traffic volume and wheel loading.

Conceptual design of unpaved gravel road is based on several references including:

- Ethiopian Roads Authority - Manual for the design of low volume roads - Part A – D.
- Southern African Development Community (SADC) - Guideline for low-volume sealed roads
- Guidelines for Human Settlement Planning and Design – Roads: Materials and Construction
- South African National Roads Agency Ltd (SANRAL) – South African Pavement Engineering Manual
- Technical Methods for Highways (TMH) – TMH 4 Geometric design standards for rural two-lane two-way roads

Gravel roads are designed with:

- Height between road crown and invert of side-drain varies from approximately 350 mm to 500 mm
  - Side drains and windrows, culverts and related storm water infrastructure suit terrain conditions varying from flat to rolling terrain
  - Cross fall on road is between 4% to 6%.
- Structural layers include natural soil (sub grade) dynamically compacted in-situ in the road box-cut overlain by selected sub-base gravel compacted in 2 x 150 mm layers thick and a 200 mm wearing course. Stabilization of the structural layers is envisaged due to:
  - Generally very low strength of natural gravel available in borrow pits that is suitable for road construction
  - Slippery conditions in wet weather
  - Scour by flowing water.

Gravel roads require substantial maintenance and frequent repair work, especially during the wet season and excessive axle loads. The procedures include:

- Resurfacing:
  - The gravel wearing course is 200 mm thick
  - Gravel loss rate is approximately 45 mm per year per 100 vehicles per day unless the surface is sealed
  - Replacement of the wearing course with new material is required every three years
  - Road work is to be undertaken by an experienced roads contractor, using appropriate earthmoving graders and compaction plant and equipment
  - Resurfacing is required over and above routine daily road maintenance.
- Daily road maintenance:

  - Maintenance work on roads and drains is suitable for community-based labour
  - Low cost due to low overheads
  - Simple contract system
  - Direct response - rapid mobilization
  - Retain skills in the community
  - Close control of work-force
  - Dispute resolution is contained within the community
  - Remuneration re-cycles within the community
  - Employment can target at poorest and disadvantaged group
  - Skills training required
  - Equipment and resources to be funded by the mine
  - Control of quality and productivity requires supervision.
- Productivity planning
  - Number of person-days per km per year = 75
  - Productivity target per person per day, assuming that daily maintenance results in light task difficulty. If maintenance is neglected, the task difficulty increases and productivity rate reduces to 50%
  - Bush clearing (light) = 450 m/day
  - Shoulder rehabilitation (manual) = 100 m/day
  - Plant grass = 100 m/day
  - Cut grass = 400 m/day
  - Blade gravel (light) 10 km/day
  - Spot repair (Selected gravel material) = 25-wheel barrows/day
  - Reshape and compact earth road camber = 70 m/day
  - Culvert cleaning = four culverts/day
  - Ditch clearing - Turn-out drains = 60 m/day
  - Ditch cleaning - Side drains = 65 m/day
  - Erosion damage repair = 7 m/day
  - Masonry repair = 7 m/day
  - Build stone scour check = 5 off/day.

7.7 Airports

The Tete International Airport (Chingozi) or TET, is 110 km by road to the Project, along the N7. The Chileka International Airport (Malawi) is 230 km by road, from the Project, whilst travelling along the N322 and M6. There are weekly flights between Tete, Beira and Maputo via the national carrier. TET can receive charter aircraft and airfreight, with TET asphalt runways over 2,500 m length.

7.8 Ports and rail

Beira is the closest port to the Project site, approximately 730 km by road (with travel time of about 12 hours one way), through Tete, Chimoio, Dondo to Beira (the southwestern route). An alternative road (northeastern route) is via Doa, Mutarara, Muanza to Beira (a route of 620 km, with travel time of more than 15 hours one way). Three routes are available to the port of Nacala, with distance varying between 1,055 km and 1,135 km (travel time range of more than 18 hours to 20 hours, one way).

This study contends that all major inbound freight cargo over the LOM (reagents and consumables) will be processed via the Port of Beira and that all major outbound freight (MREC) will also be through the Port of Beira. The Port of Beira is serviced by all major sea freight carriers.

Two railway lines connect the project area to the Indian Ocean ports of Beira and Nacala: the Sena line and the Nacala Corridor respectively. The Nacala Corridor, opened in 2017, is the preferred route to export coal from Moatize.

A detailed cost-benefit of the various logistical options both for inbound and outbound freight cargo will need to be done as part of the PFS.

7.9 Buildings

Non-plant buildings and structures will include the following:

- Diesel fuel tanks.
- Change house (mining).
- Workshops and associated offices.
- General offices.
- Explosives magazine storage.
- Emulsion receiving, storage and distribution area.
- Accommodation and catering facilities.
- Medical facilities.

Primary plant buildings will consist of the following:

- Gatehouse.
- Change house and laundry.
- Clinic.
- Canteen.
- Plant control rooms.
- Office buildings.
- Metallurgical laboratory.
- Plant workshop.
- Plant main store.
- Reagents store.
- Final product store.
- Air services building.
- Blower air building.

7.9.1 Main gate entrance

The mine entrance gate, visitor parking and security kiosk is located to the north of the planned mine. The facility comprises of the following elements:

- Containerised security kiosk
  - Container unit supported on concrete plinths and foundations
  - Entrance and exit turnstiles and fencing

  - Portal frame steel roof structure with independent column supports covering the kiosk and perimeter walkways with an eaves height of 2,700 mm and 10° roof slope.
- Parking area
  - Gravel surface parking area (1,200 m^2)
  - 30 parking bays.
- Bus terminus.
- Circulation paving.
- Access roads incorporating air-lock gates for vehicle inspections.
- Environmental toilet
  - Permanent toilet and basin inside the kiosk is commissioned when potable water and sewer pipelines are installed.
- Fencing
  - Standard fencing temporarily extending 50 m on either side of the security kiosk.

7.9.2 Diesel fuel tank depot

Based on the planned fuel consumption defined during PFS design work:

- This fuel depot will have a plan area of approximately 100 m^2.
- Diesel provides power to mobile equipment and the plant. The total usage forecast for diesel per annum will be defined at PFS in order to achieve a 7-day storage buffer.
- A full design, installation and operation of the diesel supply infrastructure will be provided by a local Mozambique distributer of liquid fuels and lubricants. The complex will be bounded by a fence with gate access for security.
- Diesel and lubricants will be transported by road directly to the fuel depot facility located at the rim of the crater. From here, the diesel and lubricants will be reticulated in pipelines to two dispensing points at the mining contractor's terrace (storage capacity of the order of 10,000 litres) and at the plant (storage capacity of the order of 40,000 litres).
- The fuel depot will house multiple diesel storage tanks with a maximum capacity of seven days supply. The tanks will be supported on a reinforced surface bed with plinths and each row will be surrounded by 650 mm high bund walls to prevent spillage. Sump systems will be installed to safely collect and dispose of any spillage.
- Bulk storage tanks for lubricants will be accommodated in the same facility.

7.9.3 Change house and laundry complex (mining)

The change house complex is situated in the crater and will comprise the following:

- Interconnected ablution blocks with change rooms, showers, toilets and basins servicing band A, B, C, D and E personnel with a separate ablution block for female employees.
- A laundry room for regular laundering of PPE, bedding and personnel's clothing.
- A boiler room which distributes hot water to the change house supplemented by roof-mounted solar water heating system.
- A radiation room 4 x 4 m with a table, desk and cupboard to monitor levels of radioactivity exposure

7.9.4 Main contractor and mining contractor laydown area

The terraces will be gravel surfaced and designed for heavy vehicle traffic. Maintenance of the terraces damaged by rutting is the responsibility of the relevant contractors. These terraces are fenced off for security purposes.

- The main contractor's laydown area terrace plan area will be defined a PFS stage:

  - Contractors to provide own facilities and security
  - Main sewer connection provided
  - Potable water connection provided.
- The mining contractor’s Infrastructure terrace plan area will be defined a PFS stage:
  - Contractors to provide own facilities and security
  - Main sewer connection provided
  - Potable water connection provided.

7.9.5 Workshops and associated stores

The mine workshops and stores will be constructed on terraces with a 6 m wide access road provided onto a paved apron slab within the workshops and stores.

Vehicle workshop

- Single storey container building covered with sheeted steel roof and partial side sheeting. Roof pitch of 10°. Supported on concrete plinths and foundations.
- Reinforced concrete ground slab (under work areas only).
- Clear span of 7 m.
- Eaves heigh of 4 m.
- Sundry vehicle inspection equipment

Infrastructure workshop

- Double storey container building covered with sheeted steel roof. Roof pitch 10 degrees. Supported on concrete plinths and foundations.
- Clear span of 10.6 m.
- Eaves height of 5.2 m.
- Reinforced concrete ground slab foundation (under work areas only).
- 3 t capacity monorail crawl beam.
- Contains heavy earth-moving equipment and general equipment wash bays with sumps for disposal or re-use of grey water.
- Contains fenced storage yard.

Plant store

- Single storey container building covered with sheeted steel roof. Roof pitch 10 degrees.
- Reinforced concrete ground slab foundation
- Portal frame span of 12.5 m
- Span between containers of 10 m
- Eaves height of 3.5 m

Plant control rooms

- Air-conditioned single storey container buildings with a sheeted roof on steel rafters and an under-roof concrete apron.

7.9.6 Mine administration office

Administration offices will consist of a complex of prefabricated containerized units with HVAC and a prefabricated containerised reception unit. The units rest on reinforced concrete plinths founded on a concrete ground slab. The office complex should contain the following facilities:

- Tea kitchen.
- Ablutions.
- Landscaped entertainment area.
- Boardroom.
- Offices.
- Storerooms.

A gravel parking facility will be located adjacent to the offices. Office and parking surface area to be designed during PFS work.

7.9.7 Medical clinic

Prefabricated containerized units will include a dispensary, treatment rooms and office as well as kitchenette and ablutions placed on a terrace with concrete apron slab foundations and pathways and parking for an ambulance.

7.9.8 Metallurgical laboratory

Air-conditioned single storey container buildings with a sheeted roof on steel rafters and an under-roof concrete apron.

7.9.9 Weighbridge

Proprietary weighbridge equipment mounted onto a concrete base. Should also contain:

- Earth ramps with wingwalls.
- Apron slab under equipment with sleeves for services.
- Elevated air-conditioned container control office foundation slab.
- Storm water management.

7.9.10 Plant substation and MCC rooms

The substation buildings and motor control centre (MCC) rooms are container structures fitted with lockable single dust proof personnel access door plus a lockable dust-proof double door to facilitate the movement of equipment in or out of the room and serve as a second emergency escape route from the room in the event of a fire or emergency.

- The doors are fitted with an approved panic release door mechanism.
- No windows are fitted to the substation buildings or MCC rooms.
- Buildings are elevated on concrete plinths to a minimum floor height of 1,200 mm above finished terrace level.

Transformer bays

- Transformer bays shall consist of hollow block masonry walls on three sides of the transformer to a height of 300 mm above the highest point of the transformer. The walls shall be constructed on top of the concrete bund walls, which are integral to the base slab and plinth supporting the transformer.
- The transformer bunded foundation shall have an oil catchment pit equipped with drainage valve and chipped stone filling, the volume of which is sufficient to capture all oil in the transformer, taking the stone filling into account.

- The transformer bay front section shall be fitted with a double gate access to prevent unauthorized entry.

Mini-subs

- Mini-sub foundations shall consist of a level concrete pad with bund wall sufficient for containment of all oil inside the transformer in the event of a spill.
- The mini-sub shall be placed on top of a concrete base and the perimeter guarded by chains to prevent entry with the necessary safety warnings and mandatory signage displayed.

7.10 Accommodation

An exploration camp is currently located on the rim of the crater, this will grow in phases during continued exploration and project development. The current accommodation is in tents, with a central mess and offices (Figure 7.2).

A single accommodation village for both the construction phase of work and for operations will be located to the north of the mine site outside of the crater, on a plateau. The village will be built in phases to meet the demand of the growing workforce. During construction, the workforce will reach a peak of several hundred workers on site. As construction is completed and commissioning activities commence, the workforce will decrease over a period of approximately six months until the completion of commissioning. At the conclusion of commissioning, the workforce will primarily be the mine operators.

The construction accommodations will likely be refurbished to meet the needs of operations, where multiple occupancy rooms will be converted to single occupancy rooms. Refurbishment of the common-use buildings is planned to ensure the facilities meet requirements for long-term use over the LOM.

The accommodation village will be connected to a hybrid power station via overhead power line connection with backup diesel generators to supply critical equipment including communications, food storage, etc. during outages.

Figure 7.2 Main office building (left); Individual tents (right)

The mine camp is to house manpower grouped by management class i.e. B, C, D and E.

- A-class workers will reside off-site.
- B-class partly skilled employees are accommodated in prefabricated air-conditioned dormitory type units without ablution facilities.
- C-class skilled and supervisory employees are accommodated prefabricated air-conditioned dormitory type units with common ablution building.
- D-class junior management are accommodated prefabricated air-conditioned units with en-suite ablution facilities.
- E-class senior management are accommodated prefabricated air-conditioned single room and lounge units with en-suite ablution facilities.

7.10.1 Main camp dining, kitchen and recreation

The following facilities are envisaged:

- Kitchen – Prefabricated unit catering for a standard menu cycle feeding 1,206 meals per day (breakfast, packed lunch and dinner) for B-E Band Employees and 82 meals per day for A Band Employees (1 per shift). The kitchen and ablution units are connected to the back-up diesel generator electricity supply system. The kitchen and ablution block discharges effluent into wastewater sewer after oils, fat and grease are trapped.
- Recreation room – Prefabricated unit with a bar/ commissary facility, reading, games, TV lounges etc.
- Courtyard – Located between the kitchen and recreation room. Paved and equipped with tables, chairs and umbrellas.
- Delivery Yard – Caters for the delivery of food and other items; e.g. bedding, additional PPE, located on the back side of the kitchen.
- Waste Yard - Management of waste generated by the camp only.

7.11 Communication

An integrated information system will be provided by the Company, including the latest operating systems enabling effective telephonic and digital communications.

A fibre optic network will be reticulated around the site, allowing for the installation of IP telephone, high-speed internet and VPN connection, and will be connected through the central node room located at the main office. Consistent and reliable site and external communications can be delivered to every location across the operational mine site, process plants and power facilities.

7.12 Construction logistics

The initial capital components will need to be transported either from premises in Johannesburg/ Maputo or from Beira port to Project site.

On a Scoping Study basis, the most probable route from Johannesburg is identified as travelling through South Africa on the N1 through Beitbridge to Zimbabwe, along the A4 to Harare (capital of Zimbabwe), then the A2 eastward through the Nyamapanda-Cochemane border post, then the N8 passed Changara district, from where the N7 is followed to Tete. Break bulk charter and transport for sensitive equipment (electrical) will follow this route.

Studies on Project cargo, clearance limits, abnormal load dimensions, height clearance, vehicle combinations and insurmountable limitations will need to be undertaken in future studies.

7.13 Container handling yard

MREC product from the beneficiation plant is bagged and loaded into shipping containers at the mine and onto trucks for transportation to the Port of Beira. This facility is also a staging point for delivery of reagents and other materials imported to the mine from elsewhere.

The container loading area is strategic infrastructure for the mine and therefore the integrity of the terrace surface, that is subjected to heavy vehicle traffic, needs to be robust. A 180 mm thick reinforced concrete ground slab is provided as a wearing surface, jointed into 5 m x 5 m panels to mitigate cracking of the concrete surface. The yard and final MREC product store will be fenced and serviced by a container office.

Storm water drains are provided to capture and direct runoff on the apron slab into catchpits and channels which discharge into the natural drainage gullies. Since the containers are sealed, the runoff is deemed to be clean water.

7.14 Ex-mine logistics of ex-plant product

For purposes of this study, it is proposed that stockpiled MREC will be placed in 1 t polypropylene, double-lined woven bulk bags at Project site, and then placed on pallets or loaded directly onto trucks. The Project site will have mobile equipment capable of loading either conventional flatbed type trucks or containers with these bags.

The bulk bags will be placed and dispatched in 20 ft standard containers from Project site to Beira. Approximately 21 bags can be dispatched per container. Containers will be trucked to Beira port and warehoused, prior to shipment. These transport arrangements are expected to result in approximately 745 truck journeys per annum (equivalent to 62 trips per month) of bagged concentrate MREC product to Beira. The containerised bags will be offloaded at Beira and then re-containerised at Beira or report straight to ocean going vessels.

7.15 Overall infrastructure costs and comments

The approximate infrastructure size and costs for the Project have been estimated. Primary infrastructure costs include:

- Power ($7.5 million)
- Access road ($7.0 million)
- Accommodation ($4.0 million)
- Sewage treatment ($2.0 million)
- Raw water dam ($2.0 million)
- Wellfield ($2.0 million)
- Stormwater ($1.0 million)
- Water treatment ($1.0 million)
- Other surface infrastructure (including gatehouse, changehouse, laundry, clinic, canteen, office buildings), of $2.8 million).

The approximate footprint of the ten primary surface infrastructure/ buildings is 10,915 m^2. As the Project advances, greater accuracy and footprint size will be estimated.

Design details will be required as the project advances to PFS stage; this will include as a priority:

- Power demand.
- Water demand.
- Detailed plans for site location.
- Detailed access road plans.

8 TAILINGS AND WASTE MANAGEMENT

The mine waste will be stored in a full-containment facility in line with the Global Industry Standards for Tailings Management (GISTM). There are two sources of waste; waste rock from overburden and mining waste, and tailings derived from the processing and beneficiation plants.

A detailed waste and tailings disposal as well as a water management plan will be developed in the next phase of the study.

8.1 Tailings disposal

8.1.1 Design criteria

All process plant waste products will likely be disposed of onto a single fully contained tailings storage facility (TSF). Pre-stripping over the mining area will provide the initial waste rock required for the containment embankment walls. As more waste is stripped over the mining areas, these waste rock embankments will be raised above the tailings level to provide solid rock embankment walls.

It has been assumed that the tailings residue will be conventionally thickened wet tailings of around 50% w/w. Supernatant water will be recycled and returned to the plant for re-use. From the preliminary sizing of the TSF, a placed dry density of the tailings is assumed as 1,4 t/m^3. The geochemical characteristics of the tailings are currently unknown.

The tailings will be placed on a 2 mm HDPE lined facility with suitably constructed under-drainage. Despite the low acid generating potential of some of the residues and the presence of carbonate rock, there will also be a component of hydrometallurgical plant waste containing residual thorium and radio-active elements, which will require safe disposal in the TSF. These hydrometallurgical residues may also be stabilised with lime / limestone ore detailed is provided in the metallurgical section of this report.

The financial model shows the required tailings dam size of 11.61 Mt for the beneficiation plant and 1.66 Mt for hydrometallurgical plant waste product. It is assumed that these wastes will be combined into a single facility of 13.27 Mt at a steady state deposition rate approximately 0.75 Mt/a, over a LOM of approximately 18 years.

There may be a potential to split the two waste streams, once planned metallurgical and geochemical work on the waste streams is undertaken, but at this stage these have been combined into a single TSF facility.

8.1.2 Characterisation of the waste streams

The bulk of the waste residue (~88%) for disposal is a flotation tailings slurry of approximately 50 % w/w solids to liquid ratio. The solution will contain some residual flotation reagents.

There is also a second combined hydrometallurgical solid residue and tailings stream (~12%), as a composite of several other smaller streams from the hydrometallurgical plant.

Although the anticipated runoff from the combined tailings and surrounding waste rock embankment is not considered to be of a high risk due to its lack of acidity and low concentrations of regulated contaminants, there is potential for the waste rock to release an elevated alkaline discharge (high pH) with minor concentrations of REEs which could pose a risk to freshwater resources.

8.1.3 Site selection

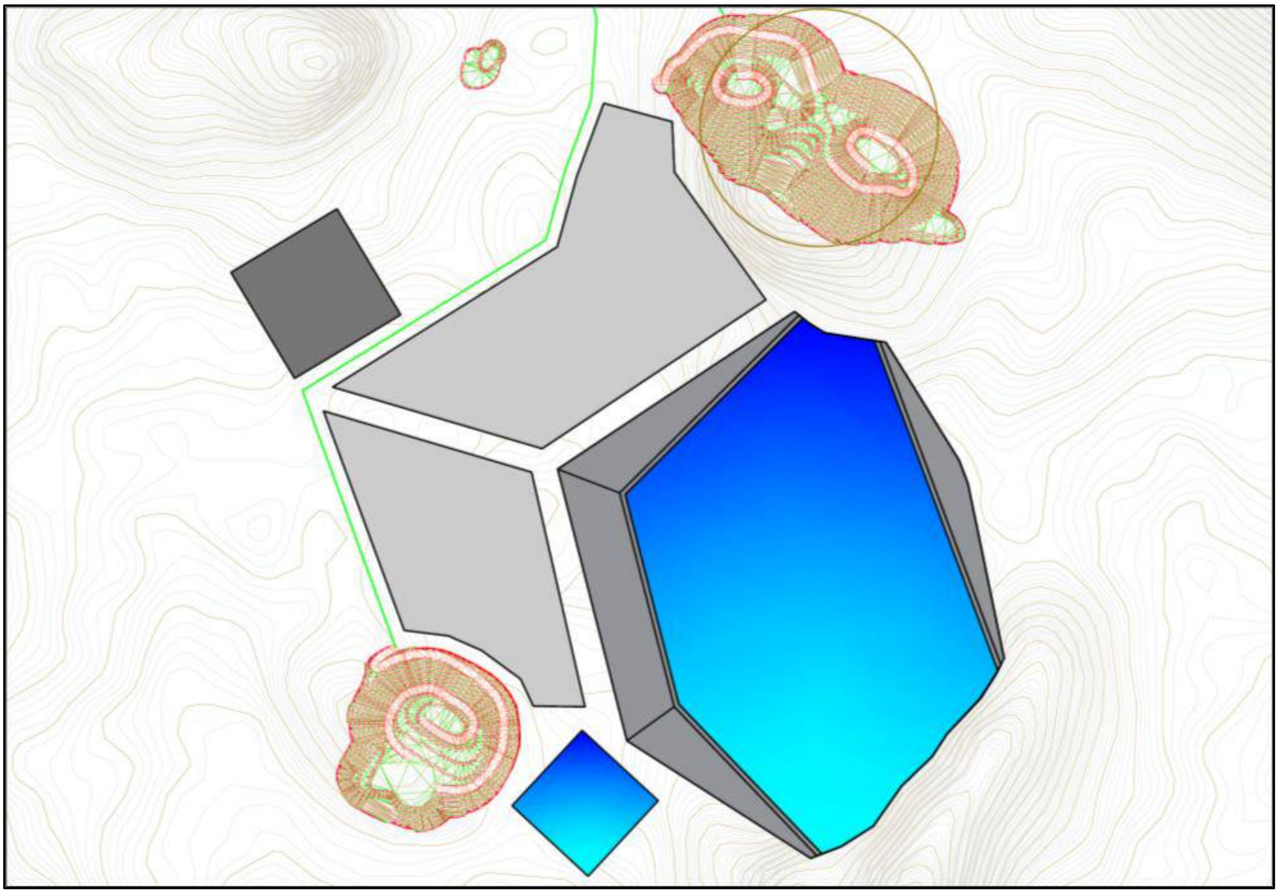
A preliminary site was selected for a storage capacity of 13.3 Mt (Figure 8.1), with a full-containment facility in line with the GISTM. The design will also need to take cognisance of the potential seismic nature of the area with the full waste containment.

All the potential sites will be further investigated during the PFS, but all would likely be contained within the perimeter of the circular layout of the crater site between the rocky outcropping hills. The intention would be to use some of these outcrop areas. The preliminary site was selected to be:

- The most economical site based on the high-level capital cost estimate.

- Lower risk site with limited infrastructure below it and close to the mining pits for the source of waste rock for the containment embankment walls.
- Sufficient space available for the construction of stormwater control dam downstream of the TSF and waste rock and ore stockpiles to the west.

Figure 8.1 Schematic layout of TSF, stormwater control dam and waste rock embankments



Note: TSF (large blue), stormwater control dam (blue square) and grey waste rock containment embankments

8.1.4 Geotechnical investigation

There is no geotechnical investigation of the TSF site at this stage, this is likely to be conducted during the PFS to follow.

8.1.5 Tailings storage facility design

The preliminary TSF design features used in the capital cost (capex) calculation comprises the following:

- A 2 mm HDPE-lined, full-containment valley TSF, constructed in a number of sequential phases or downstream lifts, following the construction of the initial starter embankment.
- The TSF embankment constructed from waste rock material sourced from the initial mining operations.
- A HDPE-lined stormwater control dam.
- Associated infrastructure, including the slurry distribution pipeline, catchment paddocks, toe drain system, underdrainage system, solution collection system, collection sumps and manholes, seepage cut-off trench, storm water diversion trenches, emergency spillways and leakage detection drains.
- A floating pump decant system to decant the supernatant tailings water and stormwater from the facility back to the process plant.

8.1.6 Design aspects of the TSF

The TSF has been designed taking cognisance of the following aspects:

- The topography, the immediate surroundings, and mine infrastructure.
- A total dry tailings storage capacity of 13.3 Mt at a deposition rate of 0.75 Mt/a over the LOM of approximately 18 years.
- Phased construction of the TSF over the LOM.

The preparatory works associated with the TSF design and capex comprise the following:

- Topsoil stripping within the TSF footprint, including the embankment wall footprint area and associated TSF infrastructure area.
- A compacted waste rock embankment with a 10 m minimum wide top-crest, an outer side slope of 1V:3H, and an inner side slope of 1V:2H, with an anchor key.
- Fine-grained material on the upstream face of the waste rock embankment wall, forming part of the total embankment wall.
- A compacted layer of suitable fine-grained material to a depth of 1.0 m beneath the TSF embankment wall to limit seepage through the waste rock embankment wall.
- A toe drain to draw down the phreatic surface within the TSF.
- Vertical and horizontal drains within the basin, in the form of a grid, above the HDPE liner, to draw down the phreatic surface within the TSF.
- Pipes at specified intervals along the perimeter of the drains, underdrains, and curtain drains, channelling the water collected by these drains into the solution pipeline.
- Seepage cut-off drain network.
- Catchment paddocks around the perimeter of the TSF.
- A gravel access road around the TSF.
- Storm water diversion channels and berms.
- Slurry distribution pipeline along the perimeter length of the TSF, with discharge outlets.
- An emergency spillway.
- Stormwater control dam.
- Leakage detection drains beneath the seepage collection dam (SCD) liner.
- A water collection manhole for the collection of leakage water beneath the liner.

8.1.7 Deposition and operating philosophy

The proposed depositional methodology for the TSF is likely to be by spigot/ open-ended discharge behind a fully contained embankment wall. This requires that each phase of the TSF embankment be built to its required height prior to commencing with that phase’s associated deposition.

During the initial commissioning stage of the Project, it is important that the tailings not be deposited directly onto the various toe drains, as this would lead to erosion and possible blinding of the toe drain system.

Tailings should be deposited into the basin of the TSF by means of an open-ended deposition technique whereby flexible hosing is utilized. Prior to the tailings reaching the various toe drains, coarse tailings should be used to cover and further protect the drains. Open-ended deposition shall continue above the covered toe drains to the final elevation of each phase.

Surface water accumulating onto the TSF emanates from the following sources:

- Supernatant slurry water on the TSF.
- Stormwater runoff from the surface of the TSF.

Supernatant water and stormwater collected on the TSF shall be decanted by a floating barge pump or turret arrangement and pumped back to the plant for reuse as process water. Given that the decant pumping systems make use of electricity to pump slurry water back to the plant, standby pumps or a diesel generator is required to adequately cope with rapid water ingress during an emergency. There will also be provision in the final design for an emergency spillway to protect the wall integrity.

8.1.8 Slope stability analysis

Slope stability analyses still needs to be carried out on a variety of possible operational and upset conditions. A deterministic and probabilistic seismic site assessment will also be required for the site during the subsequent design phases.

8.1.9 Capex and opex

The capex cost estimate was factorised from our database of costs into Q3 2023 prices. The overall TSF has an estimated capex of $54 million over the 18-year life (Table 8.1). It will be possible to further divide the capex over various design phases with future design work, to reduce the initial capital and to increase the sustaining capital over the subsequent tailings dam lifts.

With a wet disposal solution, the first phase or initial lift will require the bulk of the HDPE lining, costed at $5.46 million.

Operating costs (opex) for the operation of the facility is expected to comprise of operational management, maintenance and surveillance, mechanical equipment replacement such as pipeline and valves. Opex also includes ongoing monitoring and daily, monthly and annual inspections. Overall, a opex cost of $0.52 per placed tonne or $390,000 p/a.

Table 8.1 Monte Muambe TSF capex

Description	Value ($ million)
Site clearance, earthworks and excavations	26.30
Drainage (and gabions)	4.70
Liner and geosynthetic materials	5.46
Concrete works	2.78
Pipe work	0.76
Decant access	0.33
Safety and security	0.88
SUB-TOTAL	41.21
Preliminary and general costs (30% of measured works)	12.36
TOTAL	**53.60**

8.2 Waste rock disposal

8.2.1 Design criteria

The mining of the resource at Monte Mumbe will include the excavation of waste rock, followed by the loading and hauling by articulated dump trucks from the pit for permanent disposal. The waste rock will be used in the construction of the TSF containment embankments. The balance of the waste will be deposited in designated waste rock dumping areas in close proximity to the pits, to reduce haulage distances. The key design parameters for the waste rock disposal are summarised below (Table 8.2).

The waste rock is expected to be non-acid forming with limited release of contaminants over the long term. The waste rock contact water is expected to show high alkalinity (due to the carbonatites) with minor concentrations of REEs. The protection of ground- and surface water resources is therefore a priority.

Table 8.2 Waste rock disposal design criteria

Description	Unit	Value	Source
Total waste rock tonnes	Mt	22	Snowden Optiro
Waste rock bulk density	t/m^3	1.8	Prime Resources
Total volume of waste rock (LCM)	Mm^3	12.3	Calculation
Volume of waste rock required for the TSF	Mm^3	3.6	Prime Resources
Waste rock storage volume requirement	Mm^3	8.7	Calculation
Waste rock storage tonnage requirement	Mt	15.7	Calculation

8.3 Site selection

The positioning of the waste rock disposal area was guided by the following criteria:

- Limit the haul distance from the pits to the designated disposal/ dumping area.
- Ensure a stable landform, both during operation and post closure.
- Effective management of drainage and surface water.
- Sufficient storage capacity.
- Minimize impacts on groundwater and surface water resources.
- Avoid environmentally- and socially sensitive areas.

The area between the T1 and T4 pits was identified as the preferred site for the positioning of the disposal of the waste rock. The position is ideally located in close proximity to both of the pits. There are no significant drainage features over the footprint.

8.3.1 Geotechnical investigation

A geotechnical investigation of the waste rock disposal area is yet to be undertaken; it will be completed during the PFS to follow.

8.3.2 Layout and geometry

The waste rock will be hauled from each of the pits to the waste rock dumps, located adjacent to each respective pit (Figure 8.1). A corridor will remain between the dumps, to provide for the placement of tailings delivery- and return water pipelines between the processing plant and the tailings storage facility. The waste rock dumps will be developed in 10 m vertical lifts, with 15 m wide benches and 1V:1.5H intermediate side slopes. The overall outer side slope profile will be 1V:3H for rehabilitation.

The waste rock dumps will cover a total footprint (natural ground) of approximately 40 ha, with a final downstream height of approximately 50 m. The total final storage capacity of the waste rock dumps is 8.7 Mm^3 or 15.7 Mt.

8.3.3 Design aspects

Prepared basal layer

In order to limit the ingress of potentially alkaline seepage into the foundation and groundwater resources, the basal layer of the footprint of the waste rock dump will be ripped and recompacted to specification. The geotechnical investigation planned for future studies will confirm the underlying soil profile and compaction requirements.

Perimeter embankment

The waste rock will be deposited behind a nominal toe embankment wall which contain the material and define the footprint of the WRD. The geotechnical investigation planned for future studies will confirm the availability of material to construct the embankment.

Drainage

An underdrainage system is included on the basin to reduce seepage to the groundwater and to collect runoff from the basin in the early stages of development. The underdrainage system will comprise of slotted pipes positioned across the basin of the waste rock dumps. The pipes will be covered with a layer of selected crushed waste rock, overlain by an additional layer of protective waste rock. The placed layers of rock will function as a coarse filter and protect the pipeline against initial traffic from haul trucks depositing the material. The slotted pipes will convey collected seepage and runoff to the perimeter solution trench.

Surface water management

Runoff from the waste rock dumps will be collected in catchment paddocks, constructed around the perimeter of the waste rock dumps. The collected runoff and direct rainfall will report a perimeter solution trench. The solution trenches collect all the water from waste rock dump and convey it to a dedicated surface water containment facility. Excess water collected in the surface water containment facility, will be discharged via an emergency spillway into the downstream receiving environment.

Stormwater management

Clean storm water runoff from the catchments upstream of the waste rock dumps will be diverted with a series of diversion berms and channels. The channels and berms are likely to be earth lined.

with light, naturally established vegetation. The diverted stormwater will be dispersed prior to discharge to mitigate soil erosion.

Rehabilitation and closure

The rehabilitation activities related to the waste rock dumps will include the removal of any hardware, pushing the intermediate side slopes to 1V:3H, blend stockpiled topsoil with the waste rock on the surface of the final landform and establish vegetation. Shaping of the final landform will be to promote runoff.

8.3.4 WRD capex

The capex for the waste rock dumps has been determined through the factorisation of database costs into Q3 2023 terms. The capex is primarily comprised site clearance and earthworks with selected concrete works and drainage material. Rehabilitation and closure related costs are excluded from the estimate. The waste rock dump capex is summarised in Table 8.3.

Table 8.3 Monte Muambe waste rock dump capex

Description	Value ($ million)
Site clearance, earthworks, and excavations	1.73
Concrete works	0.11
Drainage and pollution control	0.26
SUB-TOTAL	2.09
Preliminary and general costs (30% of measured works)	0.63
TOTAL	**2.72**

9 ENVIRONMENTAL AND SOCIAL

9.1 Altona licences

Exploration activities on LPP7573L are carried out under an environmental management plan (EMP) prepared by local environmental consultancy GeoAmbiente Lda. The Company's activities were subjected to an independent Environmental Audit which was validated the National Agency for Environmental Quality Control (AQUA) of Tete Province on 24 October 2022.

As part of its Mining Concession application, the Company will prepare an EMP covering the proposed mining operations, and subsequently a Level A EIA.

9.2 Regulatory framework

Mineral exploration and mining activities in Mozambique must comply with the provisions of the Environmental Law (Law no 20/1997 of 1 October), the Mining Law (Law no 20/2014 of 18 August), and the Environmental Regulations for Mining Activities (Decree no 26/2004 of 20 August).

The Mining Law and the Environmental Regulations for Mining Activities classify mining activities in three levels, based on the scope of the activities and the complexity of the equipment used. Each level has different licensing compliance requirements.

Level A covers mining activities carried out on a Mining Concession. These activities require a full Environmental Impact Assessment (EIA), which must be prepared by an environmental specialist licensed by the Ministry of Land and Environment (MITA). The EIA process aims at producing a project-specific environmental licence.

The EIA licensing process involves:

- The preparation and submission to MITA of a set of Terms of Reference (ToR), which must include the timing and procedures for public consultation, a risk and emergency management plan, and an Environmental Management Plan (EMP).
- The review of the ToR by MITA and MIREME.
- If the EIA is approved, MITA issues an Environmental Licence within 10 days from the date of approval. The Environmental Licence is valid for the duration of the Mining Concession but must be reviewed every five years.

The holder of a Level A Environmental Licence must also submit an annual environmental management report, with the monitoring process carried out either by the concessionaire or by an independent consultant.

Level A activities also require the provision of an environmental bond to cover rehabilitation activities during the closure of the mine. The bond may take the form of an insurance policy, a bank guarantee or a deposit in cash in a bank account provided by MIREME. The value of the bond is based on an estimate of the costs of such restoration, which will be calculated during or after the active life of the project. The value of the bond is set by MIREME and reviewed every two years.

Level B covers quarrying activities, pilot projects done during the exploration phase, and mining activities carried out under a Mining Certificate. Such activities require a Simplified Environmental Impact Assessment (SEIA).

Level C covers exploration activities, and non-mechanized mining activities carried out under a Mining Pass. Level C activities require an approved EMP.

9.3 Environmental and social fatal flaw analysis

A review of the Project's main Environmental and Social risks was undertaken by local environmental consultancy GeoAmbiente Lda (Jamal et al, 2023) and is summarized herein. The review is based on the 2021 EMP and on a desktop study.

Exploration activities are covered by a simplified EIA and any future possible mining require the submission of an EMP as part of the Mining Concession application process. For mining a full EIA is required.

Comprehensive stakeholder consultation at national, regional, and local levels will be undertaken during the ESIA, and will be conducted by a combination of Mozambican specialists, technical advisors, and Altona personnel. Stakeholder consultation activities included stakeholder interviews, focus group discussions, community mapping, and public meetings. The focus during the ESIA will be on the Project's potential impacts and appropriate mitigation and management.

9.3.1 Geopolitical risk

The Project area, and the Tete Province in general, have enjoyed a long-term stability, with no political or military instability recorded in the past 30 years.

Since 2017, Mozambique has suffered insurgent attacks began in the north-eastern tip of the country, more particular in the Cabo Delgado province where the liquefied natural gas projects are being developed by TOTAL, Eni and ExxonMobil, to the extent that the onshore projects were halted in March 2021. A joint Rwanda and SADC member forces (Angola, South Africa, Botswana, Lesotho, Zimbabwe, Malawi and Tanzania) has succeeded in disabling some of the insurgent cells, and recapturing most of the Mocimboa da Praia District in Cabo Delgado. The development of the offshore project continued, with first LNG export having taken place in November 2022.

The Project is located over 700 km from the insurgency-prone areas, and no such insurgent activity has ever been recorded in Tete Province.

9.3.2 Socio-economic risk

The development of mining projects usually entails deep changes in the socio-economic fabric of affected region. The Tete Province is not new to this, with the past 20 years having seen the development of several large coal mines, which created employment and economic activity, but also significant social disturbance, including the relocation of over 2,000 households.

Identified socio-economic risks include:

- Loss of land for subsistence.
- Increased cost of life.
- Lack of benefit from employment and business opportunities due to lack of education.
- Discrepancy between community expectations and reality.

There are no human settlements inside the Project's Licence. It is therefore expected that the Project will not involve any resettlement, or loss of subsistence land. Traditional activities such as honey harvesting currently occurring at Monte Muambe will be able to continue outside of actual mining areas, which will cover only a part of Monte Muambe.

Neighbouring communities are already involved in the Project, mostly through employment of non-skilled labour. The Company will implement a corporate social responsibility (CSR) programme from the PFS stage, and an education programme to ensure that as many community members as possible can access employment, including for skilled positions, as well as business opportunities arising from the Project.

9.3.3 Cultural and heritage

Desktop literature research revealed that while archaeological sites are known in Moatize and Zobue, no archaeological site exists within the Licence area.

A cultural site located in a cavern in the western part of the basin is presently occasionally used by the local community for traditional ceremonies.

In compliance with relevant legislation, the Company will commission an archaeological survey of the Project Areas as part of the EIA's baseline study.

9.3.4 Hydrological sensitivities

The Licence is located between the temporary Lulera river about 6 km to the west, and the permanent Minjova River 10 km to the east. Temporary streams immediately north of Monte Muambe are used by the Djendje community as sources of water for human and livestock consumption, but these are isolated from the Project area by the northern part of the Monte Muambe ridge.

The inner part of the Monte Muambe ridge drains towards the south into the Minjova River, and ultimately into the Zambeze. The first downstream human settlements are located about 10 km southeast of Monte Muambe.

Currently, the Company draws water for its camp from an exploration drill hole converted into a borehole.

The Monte Muambe area, like the most of the Tete Province, is water-stressed and access to water is an issue identified by the communities living around Monte Muambe as important.

As part of the PFS, the following water-related matters will be considered and addressed:

- The availability and suitability of ground water within the Project area for industrial use (mine and plant) as well as for human use.
- The protection of surface water from any mining-associated contamination.
- As part of CSR activities, the improvement of water access conditions of local communities.

9.3.5 Ecological sensitivity

The Licence is not located in any environmentally protected area.

The area of Monte Muambe belongs to the Zambezia and Mopane Woodlands, which is subdivided into four different vegetation types. The vegetation type present at Monte Muambe is the Miombo dry deciduous mixed woodland, which occurs between 200 and 800 m in areas with rainfall between 600 and 1,000 mm. The most representative species are Afzelia quanzensis, Commiphora spp., Combretum spp., Pterocarpus brenanii, D. condylocarpon, Diospyros kirkii, D. loureiriana, Lannea schweinfurthii, Piliostigma thonningii, Terminalia sericea, T. stuhlmannii, Dalbergia melanoxylon, Sclerocarya birrea, Xeroderris stuhlmannii, Philenoptera violacea, Acacia nigrescens and Julbernardia globiflora.

Miombo woodlands are considered important for several species of avifauna, although the Monte Muambe area is not included in any Important Bird Areas (source: http://datazone.birdlife.org/site/mapsearch). Six endemic bird species were identified in the area: Agapornis lilianae, Pinarornis plumosus, Serinus citrinipectus, Agapornis nigrigenis, Hypargos margaritatos and Lybius chaplini.

A study conducted in the neighbouring Moatize area showed that the fauna is largely depleted. Animal species expected at Monte Muambe include genets (Genetta tigrina), African civet (Civettictis civetta), servals (Felis serval), northern grysbok (Raphicerus sharpei), duiker (Sylvicapra grimmia) klipspringer (Oreotragus oreotragus), and several rodents of the genus Cricetomys, Thryonomys and Grammomys. Bands of baboons (Papio sp.) occasionally visit the Licence area.

The IUCN Red List of Threatened Species database does not highlight the presence of any vulnerable or endangered plant or animal species in the Monte Muambe area.

Nevertheless, Monte Muambe is a mountainous environment, and has undergone very little anthropogenic transformations. Untouched habitats are likely to host fauna and flora species of interest. This fact contributes to the area’s ecological sensitivity. In its report, GeoAmbiante Lda provisionally classifies the area as of medium to high sensitivity and recommends that a baseline flora and fauna study be conducted as part of the EIA process for the Mining Concession.

9.4 Radiation management

The Project ore contains low levels of thorium (Th) and uranium (U). The LOM average concentrations for the bastnaesite ore are 200 ppm Th and 20 ppm U at Target 1 and 330 ppm Th and 7 ppm U at Target 4, which is favourably low compared with other rare earth deposits. The Project’s flotation tailings will contain lower levels of radioactivity because thorium and uranium are mostly associated with the rare earth minerals (and hence removed from the tailings).

The mineral concentrate produced from the Project, whilst having an upgraded Th and U content, is expected to have a specific activity well below the trigger point of 10 Bq/g and will therefore not be deemed as Class 7 Dangerous Goods for transportation purposes. Note that this concentrate does not leave the site; it is fed directly to the hydrometallurgical plant.

Radionuclides (Th, U and the decay nuclides) will be removed during the hydrometallurgy refining stage to produce a radionuclides-free MREC.

Altona will develop a comprehensive radiation management plan (RMP) and undertake regular monitoring and regulatory compliance of radioactivity levels of all activities including exploration, mining, processing and tailings disposal.

9.5 Closure and remediation

The intent for closure planning at the Project is that disturbed areas will be rehabilitated and closed in a manner to make them physically safe to humans and animals, geotechnically stable, and geochemically non-polluting/ non-contaminating. It is the Company's intent that a sustainable solution is agreed upon for post-mining land use, without unacceptable liability to stakeholders.

In addition, environmental rehabilitation will be ongoing throughout the LOM. Decommissioning activities are likely to include the following:

- Dismantling of buildings and infrastructures.
- Rehabilitating haul roads and hard stand areas.
- Ensuring access to the void left from open pit mining is restricted.
- Reprofiling slopes and top surfaces of waste rock dumps, stockpiles and TSF to ensure stable landforms.
- Revegetation of previously disturbed areas with indigenous vegetation.

10 MARKET OVERVIEW

10.1 Rare earth elements (REEs)

REEs comprise 15 chemical elements of the Lanthanide group, to which are usually added yttrium and scandium. Promethium virtually doesn't exist in nature, and scandium is absent at Monte Muambe. REE have similar chemical and physical properties, arising from the nature of their electronic configuration, which leads to a stable 3+ oxidation straight (Henderson, 1984). The ionic radius increases slightly with the atomic number. As a result, these elements usually occur together, but can also be partly fractionated by geological processes. This means that the percentage of each REE in a given deposit varies. La to Sm are usually grouped into Light Rare Earths Elements (LREE) and Gd to Lu into Heavy Rare Earths Elements (HREE). Yttrium is usually grouped with HREE due to similar chemical properties and geological association.

10.2 Occurrences

Crustal abundances of REE are summarized in Table 10.1 below. (USGS, 2014)

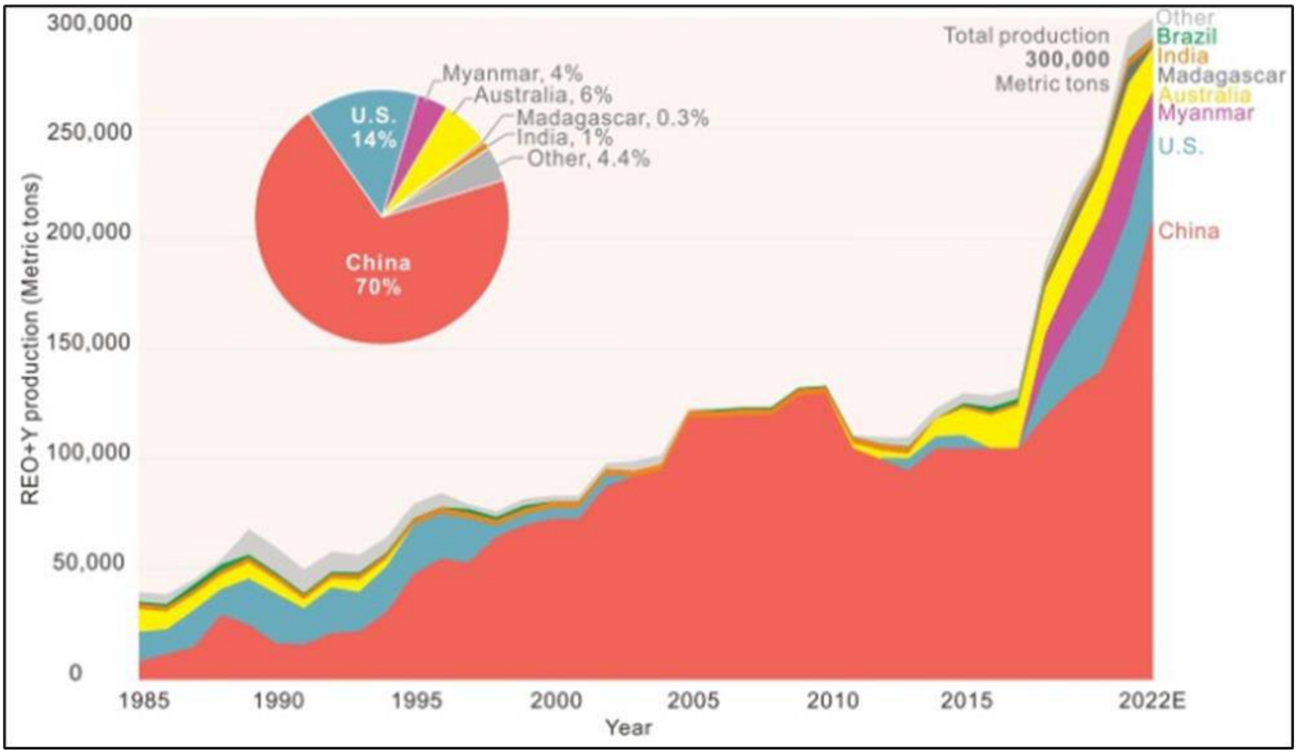
Table 10.1 REE crustal abundances

Element	Crustal Abundance (ppm)	Element	Crustal Abundance (ppm)
La	39.0	Gd	6.2
Ce	66.5	Tb	1.2
Pr	9.2	Dy	5.2
Nd	41.5	Ho	1.3
Sm	7.1	Er	3.5
Eu	2.0	Tm	0.5
		Yb	3.2
		Lu	0.8
		Y	33.0

REEs are not particularly rare, with the crustal abundance of Nd and Pr (41.5 and 9.5 ppm respectively) being of the same order of magnitude as that of Cu (50 ppm) and Pb (14 ppm). However, the occurrence of mineable REE concentrations is much more unusual. In igneous rocks, these exist only in uncommon, highly differentiated rocks such as carbonatites or granitic pegmatites. REEs can be concentrated further through various geological processes including remobilisation, weathering and erosion.

There are many REE mining projects in development across the world (Figure 10.1), however REE production currently comes from a small number of sources. According to Liu et al (2023) 86.5% of the World's REE production currently comes from carbonatite REE deposits, with the Bayan Obo mine in China accounting for about a third of the world's production. Other large REE mines include Mount Weld in Australia and Mountain Pass mine in the US. Aside from these deposits, there are also a multitude of small-scale Ionic Clay mining operations, mostly in Southeast China and Myanmar.

Figure 10.1 Global mine production of rare earth oxides for the period 1985-2022, by country



Note: The upper-left corner inset is the production percentage by country in 2022.

Source: Liu et al (2023)

10.3 Uses

REE have a wide range of uses in industry and high-technology applications including computers and cell phones, medical equipment, defence etc. These uses can be subdivided into two categories (Goonan, 2011):

- Mature markets, including catalysts, glass making and polishing, metallurgy (excluding battery alloys), and phosphors.
- High-Technology markets that have been developing over the past decades, including ceramics, battery alloys, and most importantly permanent magnets.

According to Transparency Market Research Inc, the Global Rare Earth Metals Market was valued at $10.6 billion in 2021. It is projected to expand at a CAGR of 7.4% from 2022 to 2031[1]. This means it is expected to more than double in this period. By volume, the main uses of REE are catalysts (19%) and permanent magnets (49%) (Table 10.2). However, by value, the permanent magnet market is by far the most important (>95%).

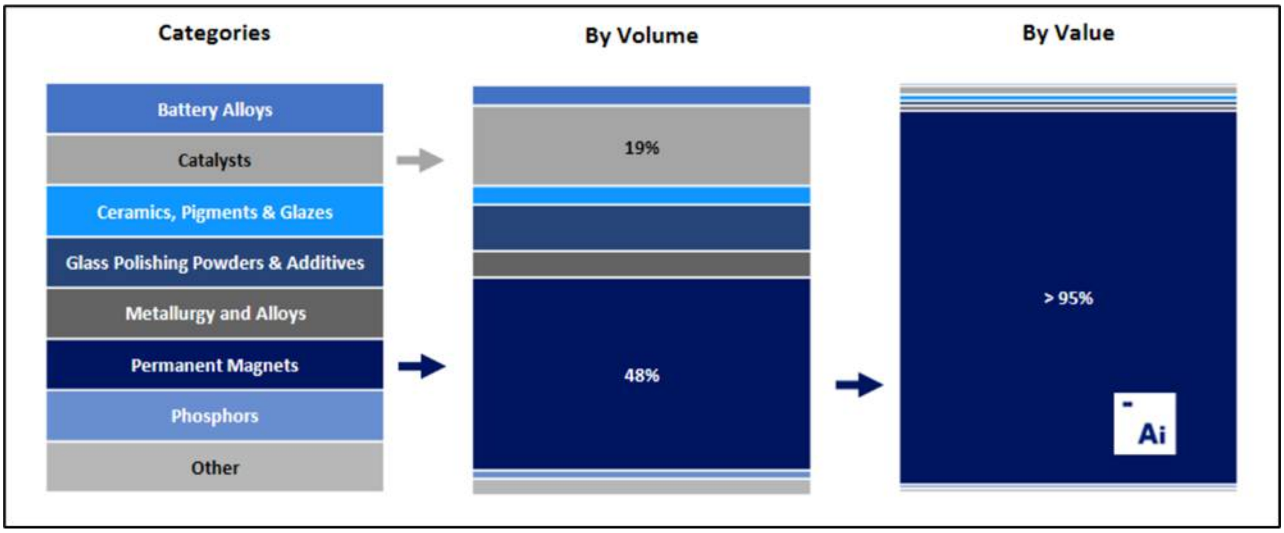
Rare-earth magnets are stronger per unit weight and volume than other magnet types (USGS, 2014), and retain their magnetic properties at high temperatures. They are usually made with four REE: Nd, Pr, Tb and Dy. These are known as the Magnet Metals. Permanent magnets are used to make generators and electric motors. This market is largely driven by the World's current decarbonization of energy sources, with the two main components being generators for wind turbines, and drive trains for electric vehicles (EVs).

[1] https://www.globenewswire.com/en/news-release/2023/06/06/2683268/0/en/Global-Rare-Earth-Metals-Market-to-Register-a-Staggering-7-4-CAGR-from-2022-to-2031-Reaching-US-21-7-Billion-TMR-Report.html

Table 10.2 Main markets for each REE (V – major market; v- minor market)

Use	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
Polishing agent		V													
Glass making	V		v	v			v							v	v
Catalysts	V	V	v	v											
Ceramics	v	v	v	v											V
Pigments		v	v								v				
Metallurgy	V	V	v	V											
Specialty alloys		V							V	V					
Battery alloys	V	V	v	v	v										
Permanent magnets			V	V	V			v	v						
Phosphors	v	v				v	v	v							V
Others	V	V	v	V	v		v			v	v	v	v	v	V

Figure 10.2 Main REE uses by volume and by value



Note: Adamas Intelligence, 2023 – figure used with permission

10.4 Pricing

Each REE has its own market price. The proportion of REE found in a given deposit depends on the geology of this deposit and is characterized by the ratio between abundant low-value elements such as Ce and La and rare high-value elements such as Lu, Dy and Tb.

As a result, each REE deposit has its own basket price. REE prices are currently largely controlled by China.

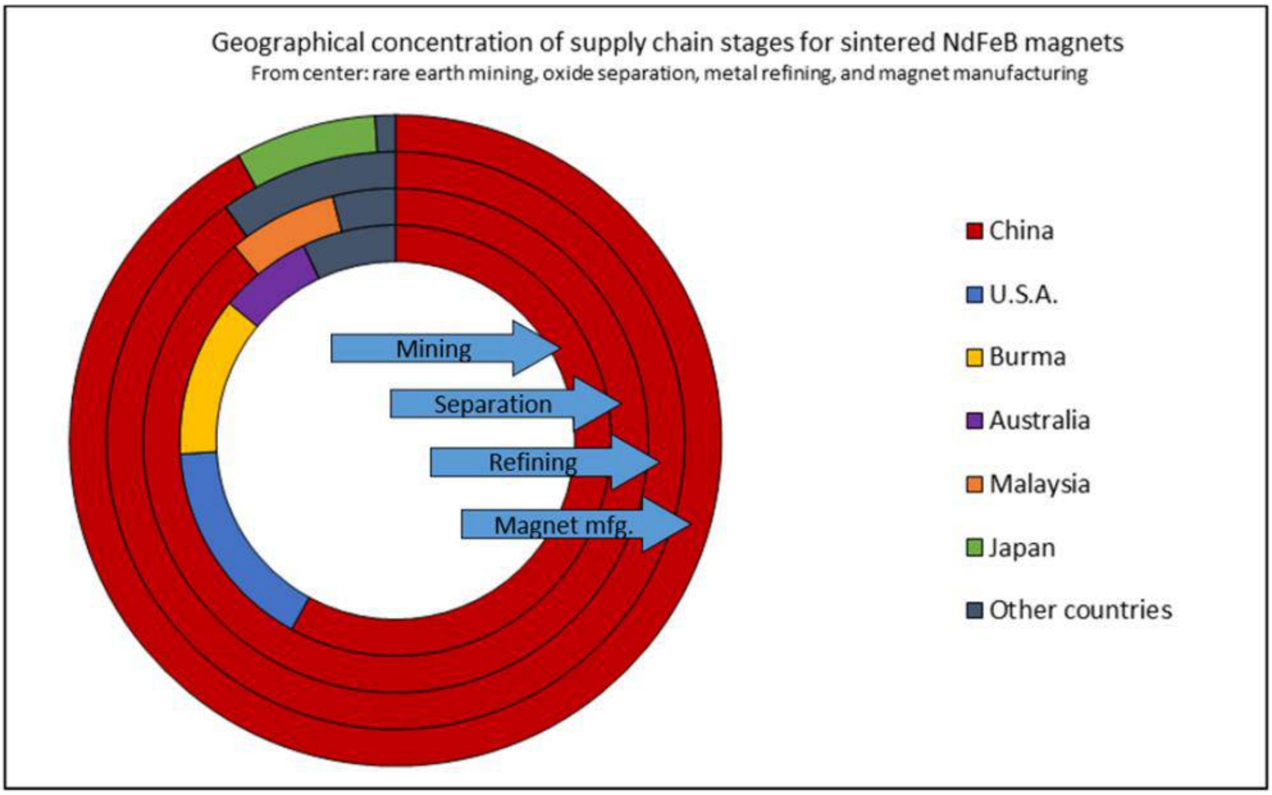
10.5 Value chain

The Magnet Metals value chain is summarized in Figure 10.3. It involves:

- Excavation.
- Production of a concentrate.
- Concentrate cracking and/or leaching to produce a mixed REE product such as a MREC.
- Separation and refining to NdPr Oxide and other rare-earths oxide products.
- Converting oxides to metal.
- Alloying to magnet manufacturers standards.
- Magnets manufacturing.

While 30 to 40% of REE are currently mined outside of China, the downstream part of the value chain is firmly controlled by this country.

Figure 10.3 Simplified value chain for Magnet Metals



Source: Smith et al, 2022

10.6 Criticality

REEs are considered as Critical Minerals because they are critical to:

- The manufacturing of wind turbines and EVs essential to the green energy transition, and which is currently concentrated in China, posing supply risks.
- Strategic high-technology applications in the defence and communications industries.

In the past years, western countries have taken steps towards reducing the reliance on China for Critical Minerals including REEs through devising policies, enacting legislation, and supporting the development of independent supply chains. This has worked effectively for the resource/ mining portion, but the downstream part of the REE chain is still over-reliant on China.

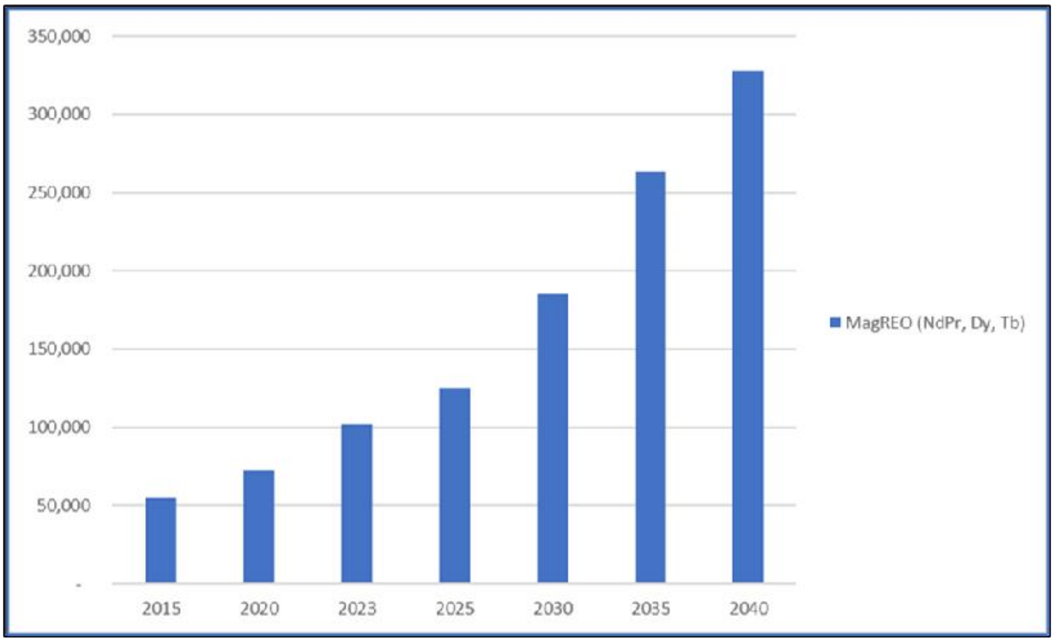
10.7 Forecasted demand and prices

Adamas Intelligence (2023) forecasts the demand for magnet metals to rise at a compound annual growth rate (CAGR) of 7.1% through 2040 (Figure 10.4). While the e-mobility and wind turbines still represent a relatively small proportion of the consumption of permanent magnets, the growth of these two market segments is expected to drive 50% of the magnet metals demand between 2030 and 2040.

At the same time, new upstream and downstream rare earths production capacity is expected to grow at a CAGR of 5.2%, insufficient to fill the projected deficit between supply and demand. The NdPr oxide supply deficit is expected by Adamas Intelligence (2023) to rise to 90,000 t p/a by 2040 (Figure 10.5).

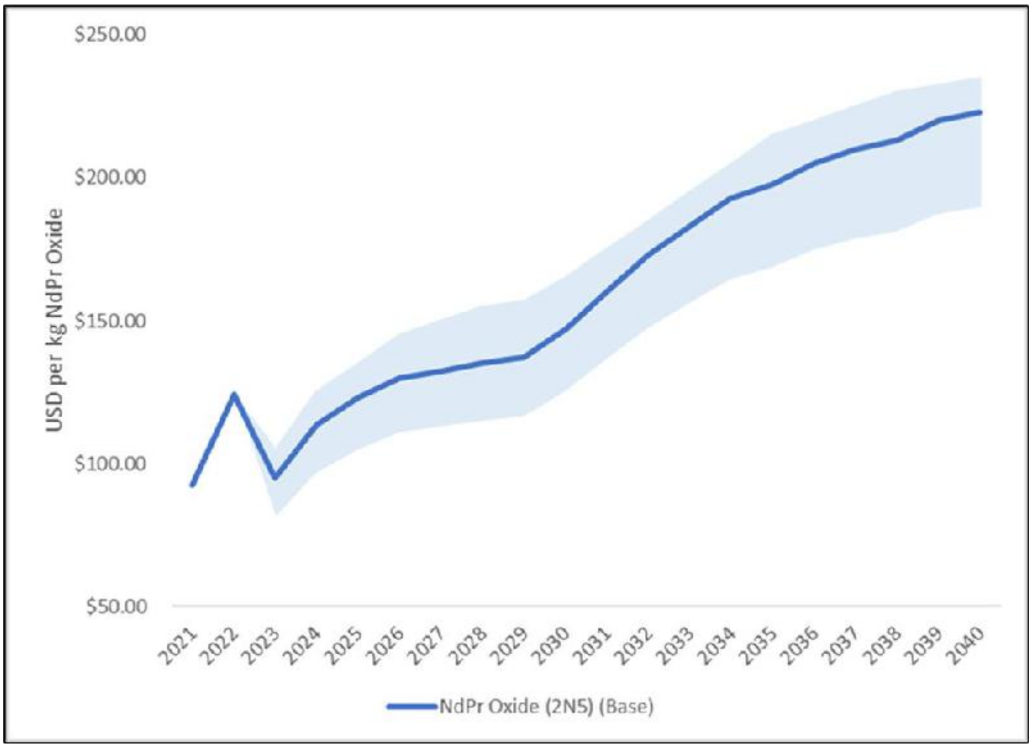
These market fundamentals are expected to sustain a growth of magnet metals oxides prices at CAGRs ranging from 5.1% to 5.2% through 2040.

Figure 10.4 World magnet metals historical and forecast demand



Note: Adamas Intelligence, 2023 – figure used with permission

Figure 10.5 World NdPr oxide forecast prices



Note: Adamas Intelligence, 2023 – figure used with permission

10.8 Monte Muambe MREC pricing assumptions

The pricing of the MREC product from Monte Muambe takes into consideration the following assumptions:

- Adamas forecast (Adamas Intelligence, 2023) 2024-2040 low case, base case and high case scenarios.
- Value for other elements $22.2/t oxide.
- Payability in MREC: 90%.

11 COSTS AND ECONOMIC EVALUATION

11.1 Introduction

Snowden Optiro has undertaken a real financial model for the Project. The base date for all financial inputs is 1 September 2023. The financial model start date is 1 October 2023, with quarterly and annual reporting over LOM. Nominal values have been estimated for the purposes of income tax and working capital exclusively. All values reported in this section are real; and all diagrams and tables have been generated from the financial model. ROM material and mineralisation are used interchangeably in this section.

11.2 Basis of estimate

The financial model has been undertaken in United States dollars ($), with the following exchange rates applied:

- ZAR19.00: $1.
- Mozambiquan metical (MT) 63.25: $1.

The capital costs were priced as of Q3 2023. No cost adjustments were undertaken. The following assumptions were made in the preparation of the opex and capex estimate:

- The LOM is 18 years.
- Only revenue associated with four primary REOs (praseodymium, neodymium, terbium and dysprosium) have been included in the financial model. No revenue has been attributed to the 13 remaining REOs
- A mining contractor will be appointed
- Landed diesel costs of $1.53/ litre have been applied in the financial model. Bulk fuel storage costing and installation will be undertaken by the primary bulk fuel supplier
- All unskilled and semi-skilled workers will be recruited locally. At least 50 % of the skilled workers will be recruited locally. Temporary accommodation to be provided for 75% of staff and contractors during construction. Permanent accommodation will be provided for 90% of the permanent staff and contractors.
- There will be no shortage of skilled trades workers throughout the entire construction phase, including the early works phase. Hence, there is no provision for salary increases potentially necessary to attract skilled trades workers.
- All employees accommodated in the camp will be provided with three meals per day.
- All other workers will be provided with daily lunch while on site.
- The camp will be located within a maximum of 30-minute walking distance from any working point for the whole duration of the Project implementation.
- There will be a smooth transition between the various project implementation phases; and no work disruption resulting from inadequate accommodation and/or catering services.
- The project schedule for the process plant is estimated to have a duration of 24 months from project award. This excludes prior construction of the access road.
- It is assumed that the bore wellfield will generate sufficient water supply for the Project. Treatment of abstracted water is required prior to use in plant and as potable water.
- Grid power will be made available at least three months before the start of hot commissioning of the beneficiation plant and hydrometallurgical plant. Diesel powered generators will be used during planned plant construction. A unit rate of $0.075/ kwh has been applied.
- The construction site will be accessible 24 hr/d and 7 d/week with adequate safety supervision.

- The construction contracts will be of the unit-rate type, cost-plus type, or lump-sum/turnkey type; the estimate does not allow for construction contracts of the time-and-material type.
- All the contractors will provide their own administration offices for the full duration of the construction phase.
- There will be no rework to field-erected and installed equipment and material, resulting from a quality assurance/quality control (QAQC) inspection.
- Residual thorium and uranium management has been included under process opex.
- It is assumed that no resettlement is required.

A number of technical design documents formed the basis of the capex estimate, including:

- Indicative, planned production schedule and LOM processing plan.
- Pit designs.
- Select mining equipment lists.
- Select process plant design criteria.
- General layouts of the process plant and related infrastructure.
- Process flow diagrams and process plant equipment data sheets and lists.
- In-house databases.

11.3 Exclusions

The following exclusions are applicable to opex and capex:

- Escalation beyond the base date.
- Duties.
- Schedule delays (associated with scope changes and labour disputes).
- Currency fluctuations.
- Force majeure.
- Financing costs.
- Contingencies.
- Marketing and selling fees.
- General management fees..

11.4 Production, revenue and metal prices

A mine schedule has been undertaken by Datamine and reviewed by Snowden Optiro. Proposed ROM steady state production of 0.75 Mt/a is reported for a mine life of 20 years. Snowden Optiro then applied a tail-cut, which has reduced the Project mine life to 18 years and is referenced accordingly as the LOM in this report.

The T1 and T4 deposits will be mined, each with their own pit. The two pits are approximately 1050 m metres apart at their closest point. A 1.5% TREO cut-off has been applied, with a Mineral Resource TREO averaging 2.42% (before dilution) over the LOM and a TREO ROM grade of 2.30% after dilution. Some 13.50 Mt of mineralisation is planned to be mined (less mining losses, plus dilution) and 21.91 Mt of waste over the LOM, with a LOM stripping ratio of 1.62.

All ROM mineralisation reports to the beneficiation plant. First stage planned recoveries are 60%, which will produce a concentrate with 10% TREO. This concentrate product will report to the on-site hydrometallurgical plant, where planned recoveries of 80% will be realised. A refined, final MREC product, with planned purity of 55% TREO, will be produced by the hydrometallurgical plant. The MREC will be stockpiled, bagged in 1 t bulk bags and then transported in 20 ft containers (by road) to Beira port. At the port the bags will be unloaded, warehoused and then loaded onto ocean going vessels.

There is minimal free dig material (less than 0.01% of mineralisation; and less than 1% of waste) over the LOM. Further study work needs to be undertaken to understand how mined oxide, transition and fresh material will report to the beneficiation plant.

Planned steady-state is reached in Year 1 of mining production.

The rare earth proportions in sold MREC are as follows:

- Praseodymium – 3.78%.
- Neodymium – 10.01%.
- Terbium – 0.08%.
- Dysprosium – 0.43%.

Long-term CIF (China) metal prices calculated as the average of Adamas Intelligence forecast (Adamas Intelligence, 2023) for the period 2024-2040 low case scenario have been applied as follows:

- Praseodymium oxide price of $148,000/t.
- Neodymium oxide price of $156,000/t.
- Terbium oxide price of $1,937,000/t.
- Dysprosium oxide price of $440,000/t.

Gross revenues total $3,670 million over LOM. Neodymium and praseodymium comprise the bulk of planned gross revenues (86%) along with dysprosium and terbium (14%); no value has been ascribed to the other 13 REOs, primarily cerium and lanthanum. A payability of 90% on the four primary elements in the sold MREC has been applied (Figure 11.1).

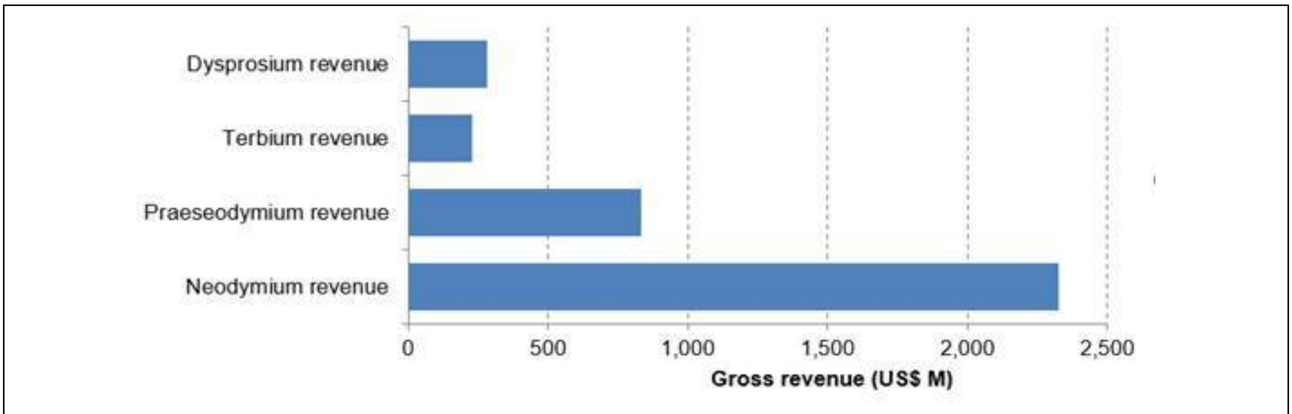
It is assumed that all MREC is sold and dispatched immediately, post-production. Benchmarked studies indicate overall payabilities would typically be 70% for unrefined REO product.

For the CIF pricing reported above, benchmarked CIF, port and warehousing charges have been applied, from the port of Beira (FOB) to China.

Net revenues include a State royalty of 3% on gross revenues;and payabilities of 90% on MREC product sold. Total MREC produced is 270.7 kt over LOM or 15.0 kt p/a, with an equivalent contained TREO volume of 148.9 kt over LOM or 8.3 t p/a. Net revenues total $3,193 million over LOM.

No provision has been made for marketing or selling fees.

Figure 11.1 Project gross revenue over LOM



11.5 Operating expenditure (opex)

The purpose of the opex estimate is to provide operating costs to an accuracy of +35% to −30% that can be utilised for the economic analysis of the Project.

The Project's annual opex estimate of production consists of the following:

- Mining operating costs benchmarked and factored by Datamine Africa (Pty) Ltd (Datamine) and Snowden Optiro.
- Beneficiation plant, hydrometallurgical plant and TSF operating costs benchmarked and factored by Met-Chem Consulting (Pty) Ltd.
- On-mine G&A or shared costs of $6.3 million p/a; benchmarked by Snowden Optiro.
- Off-mine logistics. Costs sources provided by Snowden Optiro and Altona management.

The basis of estimate and exclusions are referenced in section 11.2 and 11.3 respectively. No allowances have been applied to opex. Key project drivers are diesel, grid power, concentrate local transport by truck, concentrate transport CIF and reagents.

The planned LOM opex and unit opex is shown in Figure 11.2, Figure 11.3 and Table 11.1. Planned total LOM opex is $1,519 million, with processing opex accounting for more than 74% of total opex.

Figure 11.2 Planned LOM opex for the Project

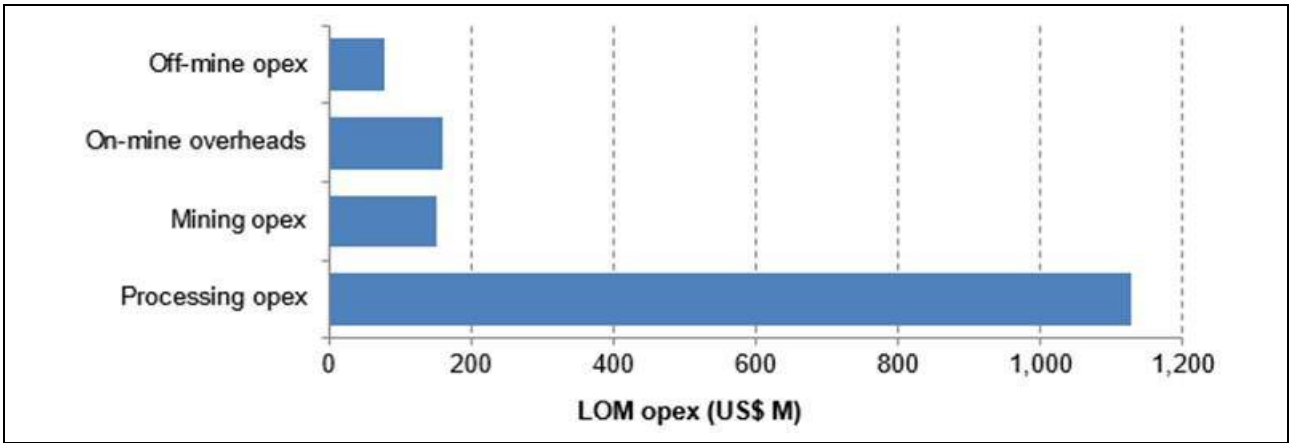


Figure 11.3 Planned opex unit cost (per t mineralisation) over the LOM

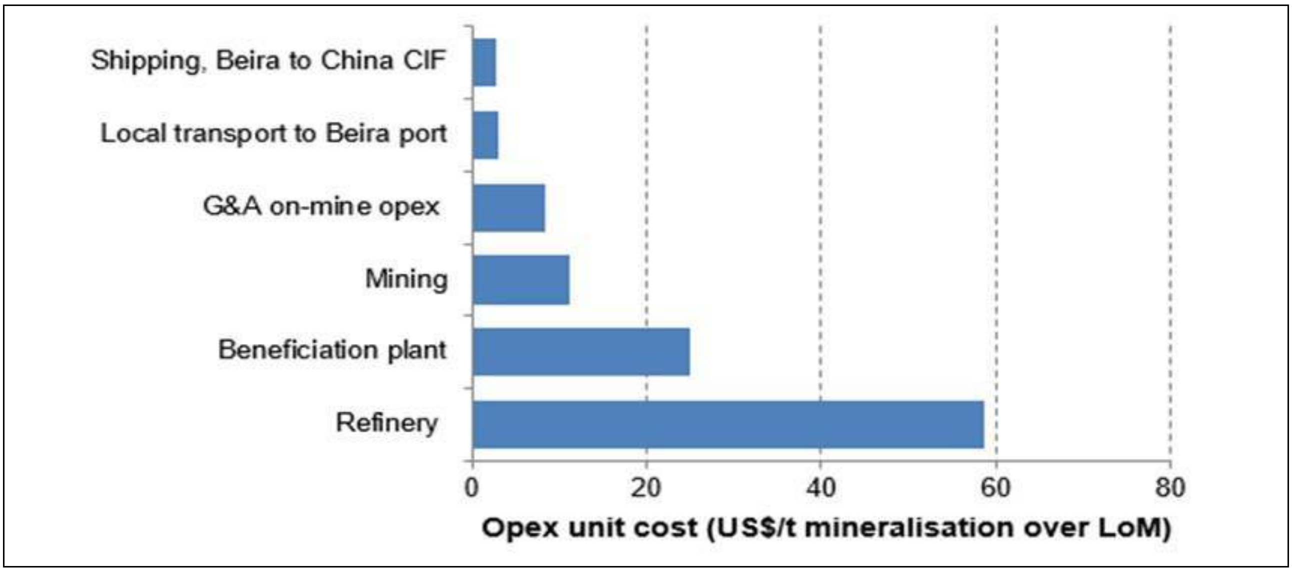


Table 11.1 Planned Project unit opex

Operating cost item	Unit	Mineralisation tonnes	Material moved	TREO in final carbonate	Final carbonate produced
Mineralisation drill and blast	$/t	4.27	1.62	386	212
Stockpiling rehandle	$/t	0.30	0.11	27	15
Waste drill and blast	$/t	5.74	2.18	519	286
Mining overheads	$/t	1.00	0.38	91	50
Beneficiation plant	$/t	25.06	9.53	2,267	1,247
Hydrometallurgical plant	$/t	58.64	22.30	5,305	2,917
Environmental and closure cost	$/t	1.50	0.57	136	75
G&A on-mine opex	$/t	8.37	3.18	757	416
Bulk infrastructure and site maintenance	$/t	2.01	0.76	181	100
State surface tax	$/t	<0.01	<0.01	<0.01	<0.01
Off-mine transport to Beira Port	$/t	2.80	1.07	253	139
Off-mine transport, Beira to China	$/t	3.12	1.18	282	155
Total unit opex	**$/t**	**112.82**	**42.91**	**10,205**	**5,613**
Volume	Mt	13.47	35.41	0.15	0.27

11.5.1 Mining operating costs

Snowden Optiro has applied unit mining rates for ROM material and waste, and further divided these into free dig and drill and blast material, as well as accounting for swell factors. A drill and blast rate of U$4.26 t/ mineralisation and $3.53/t waste has been applied in the financial model. Stockpile rehandling costs ($0.30/t mineralisation) and a charge of $1.00/t mineralisation) for grade control, Owners Team contractor management and dewatering has been applied in the financial model. A dilution of 5% and mining loss of 5% has been applied to ROM material. In the financial model, all mined material will be drill and blast.

Mining costs are defined as the costs of all ongoing mining from the time that mineralisation is mined and includes the cost of mining ROM and waste material from the open pit, including the cost of manpower and consumables; and the costs of maintaining the surface infrastructure. A mining contractor will be appointed to conduct open-pit mining and a leasing arrangement will be adopted for the mine fleet.

The mining opex estimate includes the following items:

- Mining contractor’s costs.
- Mining contractor’s overhead costs and charges.
- Fuel costs.
- Grade control drilling costs.
- Owner’s team manpower costs (mining).

11.5.2 Process operating cost

Process plant costs have been benchmarked and factored in relation to two other rare earth studies undertaken recently in southern and eastern Africa. Beneficiation plant opex has been estimated on a ROM unit basis, with hydrometallurgical plant opex estimated on an annual basis.

Processing costs include the cost of processing the ROM material to saleable products, including the cost of manpower (plant operating and maintenance labour), consumables and reagents, maintenance and bulk supply.

A fuel farm with emergency diesel generator sets has been sized for an assumed limited duration of grid power unavailability. The backup diesel generators are configured to operate in a prime operating mode. Backup generators have been allocated to allow for an n+1 redundancy.

A high-level labour schedule was developed assuming a six-weeks-on and three-weeks-off roster for expatriate personnel and two 12-hour, two-shift cycles for Mozambican national personnel.

The salaries for expatriate personnel and Mozambican national personnel were based on remuneration rates in line with market rates internationally and in Mozambique, whilst considering the availability of qualified and unqualified labour in the mine locale. Expatriate personnel will be employed in some managerial and supervisory positions. The rest of the positions will be occupied by Mozambican nationals local to the mine site.

Plant maintenance and supplies costs refer to the costs of operating spares, lubricants and other maintenance-related consumables for the plant. It has been assumed that the plant will experience a moderate amount of wear. The annual maintenance cost is estimated by multiplying the total initial capital cost with a maintenance factor that has been determined by previous projects and observations on running plants. In addition, process sustaining capital has also been applied in the financial model and discussed in Section 11.6.2.

The opex associated with the TSF and water treatment has been estimated at $0.75 million per annum over the LOM, based on a tailings dam design and costing studies undertaken.

11.5.3 Shared / overhead services

The cost of shared services for the support of the operation, including the cost of on-site labour, infrastructure, camp costs and bulk supply. The following costs were excluded from process and included under G&A operating costs:

- Camp food and catering facility.
- Expatriate travel.
- Central services labour.
- HR, safety, training and IT.
- Software, insurance and consultants' fees.
- Select light vehicles for process and shared services personnel.

Additional shared services costs include:

- Infrastructure, bulk services and site maintenance.
- Environmental and closure costing.

Of the shared services, G&A opex ($112.7 million over LOM), infrastructure and site maintenance ($27.0 million) and closure costing ($20.3 million) are the primary components.

11.5.4 Off-mine costs

Off-mine costs include bagging, warehousing and despatch of MREC. Containers will be received at Project site, loaded with 21 bags per 20 ft container and then despatched to Beira port. A cost of $139/t MREC has been allocated for bagging, loading, unloading, on-site warehousing and transport to Beira port.

A total CIF cost of $155/t MREC has been allocated for port warehousing, port charges, stevedoring, shipment from Beira to China, insurance and paperwork.

11.6 Capital expenditure (capex)

The Project's capex estimate breakdown with associated responsibilities consists of the following:

- Snowden Optiro and Datamine – Mining.
- Met-Chem Consulting (Pty) Ltd – Beneficiation plant, hydrometallurgical plant and process associated infrastructure.
- Prime Resources (Pty) Ltd – Tailings and mine waste management facility.
- Ritchie Midgley Consulting Engineers (Pty) Ltd – primary bulk and surface infrastructure.

- Altona and Snowden Optiro – Owner's pre-production costs and logistics.
- Snowden Optiro – Environmental management, rehabilitation and closure.

The basis of estimate and exclusions are referenced in section 11.2 and 11.3 respectively. No contingencies have been included in any capex estimates.

11.6.1 Initial capital estimate

The initial capex estimate consists of the direct and indirect costs, including Owner's costs and contingency costs, to be expended during the implementation phase up until planned commercial production.

The initial and sustaining capex prepared for this report qualifies as a Class 4 estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11. The accuracy of the initial capital estimate is assessed at +35 % to −30 %.

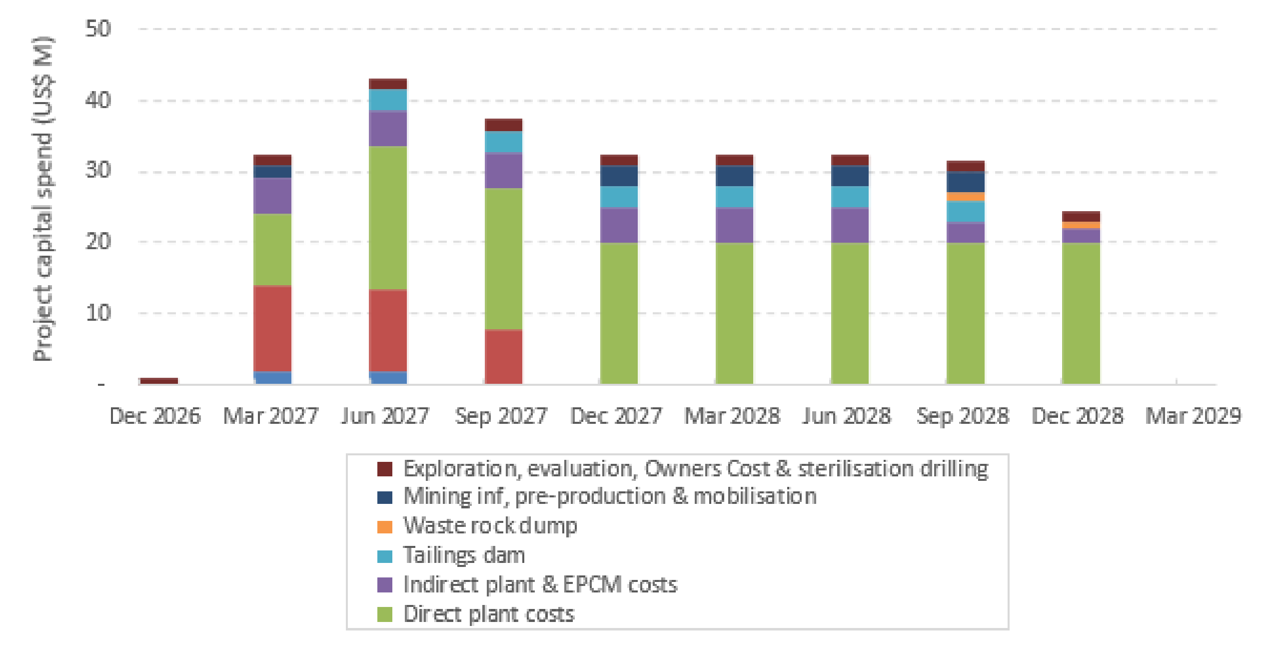
The total initial and sustaining capital for the Project was estimated to be $339.3 million, which includes project execution, EPCM, contingency and sustaining capital costs. Initial capital is estimated to be $276.3 million and includes all capex over the period October 2023 to December 2028. The initial capital is summarised in Table 12.1. Phasing of planned initial capex is shown in Figure 11.4 (exploration and study capital costs in previous quarters have been excised).

Table 11.2 Initial capital summary

Initial capital item	Value ($ M)
Project mobilisation & camp construction	4.0
Bulk & other infrastructure	31.3
Direct plant costs	150.0
Indirect plant and EPCM costs	35.0
Tailings dam	18.0
Waste rock dump	2.0
Mining inf, pre-production and mobilisation	14.0
Exploration, evaluation, Owners Team and sterilisation drilling	22.0
Total initial capital	**276.3**

Note: EPCM – Engineering, procurement, construction management; rounding has been applied to select initial capital items.

Figure 11.4 Planned initial capital phasing per primary capital item



Bulk and other infrastructure includes the following primary items:

- Power plant ($7.5 million).
- Access road ($7.0 million).
- Mobile plant, cranes and equipment ($4.0 million).
- Raw water dam ($2.0 million).
- Wellfield ($2.0 million).
- Internal roads ($2.0 million).
- Sewage and water treatment (combined cost of $3.0 million).

Direct plant costs comprise the following primary items:

- Machinery and equipment ($51.6 million).
- Earthworks, civil works, structural steel, plant infrastructure and platework ($38.5 million).
- Electricals, instrumentation and E&I installation ($22.4 million).
- SMPP installation ($21.9 million).

Indirect costs comprise the following primary items:

- Commissioning, critical and operational spares ($4.9 million).
- EPCM, associated insurances and guarantees ($26.5 million).

Mining infrastructure and mobilisation includes:

- Mining contractor mobilisation fees ($4.0 million).
- Mining infrastructure, comprising explosives, magazine, lighting, workshops, wash.
- No mining fleet has been provisioned for. A mining contractor will purchase and/ or lease all mining fleet for the Project. Diesel tank storage will be installed by the mining contractor, with an equivalent 30 days consumption to be stored.

Owners' costs comprise:

- Exploration, drilling and sampling programmes, testwork, studies ($10.0 million).

- General Owners Team costs ($6.0 million).
- Sterilisation drilling costs ($2.0 million).
- Customs, DUAT purchase, other ($6.0 million).

11.6.2 Sustaining capital

The sustaining capex estimate comprises direct costs, indirect costs, Owner's costs and contingency costs, and covers all the costs to be expended during the period starting at commercial production and extending until the end of the planned LOM. Sustaining capital over LOM totals $63.0 million, comprising three tailings lifts ($35.0 million), ongoing process capital ($27.0 million) and WRD expansion ($1.0 million). No provision for mining sustaining capital is reported. Site and bulk infrastructure maintenance charges have been included under opex.

11.6.3 Working capital

Working capital is defined as those fixed and variable costs incurred by the mine from commissioning to the point where the mine is cash flow positive, and the revenue from concentrate sales can pay for the mine's operational costs. Working capital has been benchmarked, by estimating a ramp-up period (period for plant to reach design production capacity) of six months. The following costs were considered:

- Operating costs for the whole operation, i.e. mining, process plant, and waste management facility
- General and administration costs.
- Mining and process plant assay costs.
- Stockholding costs.

For the LOM, debtors days of 30 days has been applied, creditors of 30 days (mining and process opex) and 15 days on inventories (select mining and process opex).

11.7 Taxes and royalties

A production tax or royalty is payable based on the value of the mineral extracted, with an applicable royalty of 3% for other minerals. This value of the mineral extracted results from the sale price of the previous consignment of the respective mineral or, if the mineral has never been sold, its market value. Production tax is to be paid at the end of the month during which the mineral was extracted. Total State royalties over LOM is $110.1 million and have been included under net revenues.

In the future, the Company may apply for a 50% reduction in this royalty if further downstream manufacture is undertaken and/ or the ex-plant product is used in the development of local industry.

A surface tax is payable. The current rate is 210 MT/ ha for prospecting licence (in Year 7 of renewals). Snowden Optiro has applied the 210 MT/ ha rate over LOM for the Mining Concession for completeness. The LOM value of less than $0.01 million is reported for surface tax.

A windfall profits tax (WPT) is referenced in Mozambican legislation. Mining Concessions or mining certificates with a pre-corporate income tax net return in excess of 18% are subject to a windfall profits tax levied on the accumulated net cash flow. The statutory rate of the windfall profits tax is set at 20%. Snowden Optiro has not applied a WPT in the financial model.

A corporate tax of 32% on cashflows (after the applied WPT) has been applied in the financial model. Total corporate tax over LOM is $372.5 million.

Provision has been made under Owners costs, for customs and duties; although there is a strong likelihood that no customs will be payable during the initial years of construction, ramp-up and first two years of steady-state production.

No government free carry has been applied to the financial model.

No capital gains, withholding or transaction tax has been applied.

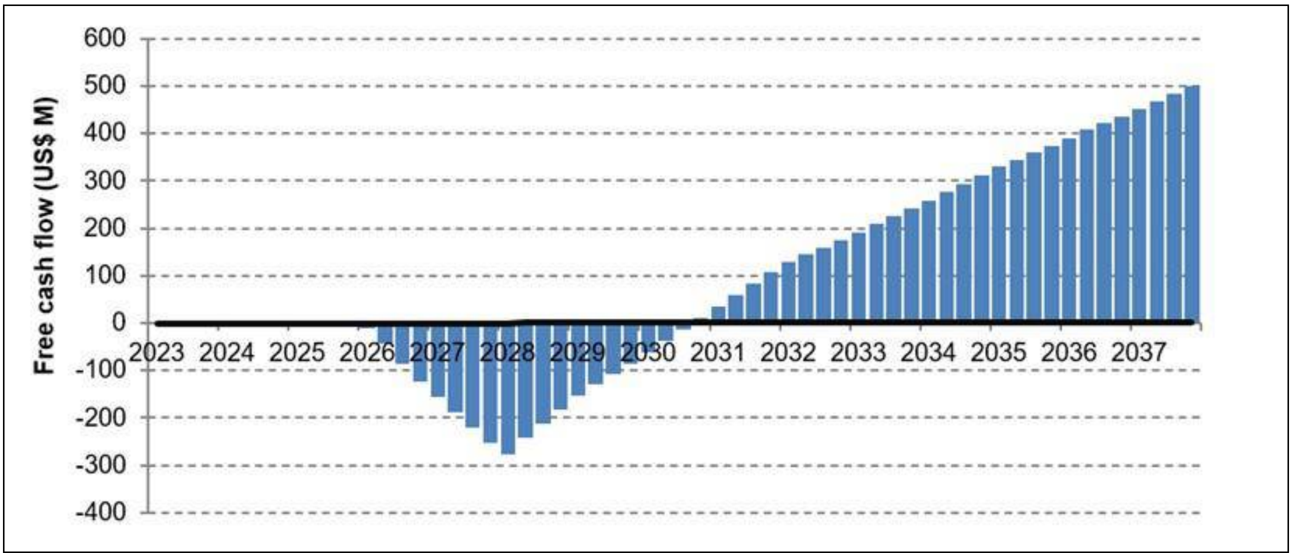
Snowden Optiro is not aware of any municipal fees or rates that are to be applied.

11.8 Net present value (NPV) and internal rate of return (IRR)

The NPV of the Project is $283.3 million, based on a real discount rate of 8% and using long-term REO prices reported in section 11.4. An NPV of $149.6 million is reported using a real discount rate of 12%. A post-tax IRR of 25% and a payback from the construction start date of 4.5 years, and a payback from first TREO production of 2.5 years is reported. An operating cashflow margin of 42% is noted. Project earnings before interest, tax, depreciation and amortisation (EBITDA) would effectively be operating cash flows (no capital expenditure, tax, interest, depreciation nor amortisation expenses have been included). Operating cashflows would include all realisation costs, on- and off-mine expenses and royalties. The planned LOM EBITDA will be $1,674 million; and planned annual EBITDA is $93 million.

Planned cumulative free cashflows over LOM are shown in Figure 11.5.

Figure 11.5 Planned cumulative free cashflow over the LOM

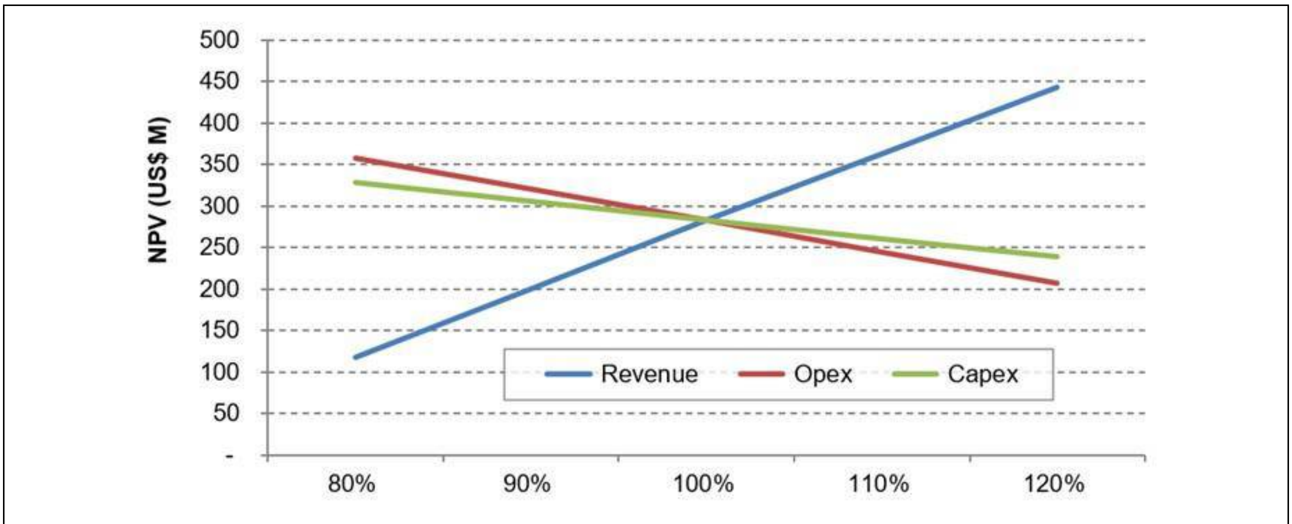


Source: Snowden Optiro, 2023

11.9 Sensitivity analysis

Using an NPV of $283.3 million with an applied real discount rate of 8%, the Project is most sensitive to revenue (price, recovery, grade and exchange rates), less sensitive to opex and least sensitive to capex (Figure 11.6). The sensitivity analysis shows that the Project is more sensitivity to capital than other benchmarked projects.

Figure 11.6 Sensitivity analysis for the Project



Source: Snowden Optiro, 2023

11.10 Summary of key Project parameters

A summary of key Project parameters is shown in

Table 11.3.

Table 11.3 Forecast key Project parameters

Parameter	Unit	Value
Ore processed	Mt	13.5
TREO ROM grade (after dilution)	%	2.30%
MREC produced	Kt	270.7
Initial capex	$ M	276.3
Sustaining capex	$ M	63.0
Opex LOM	$ M	1,519.3
Opex per sold MREC	$/t MREC	5,612.6
Gross revenue LOM	$ M	3,670.2
Net revenue LOM	$ M	3,193.1
EBITDA LOM	$ M	1,673.8
Gross revenue per tonne MREC	$/t	13,558.4
Net revenue per tonne MREC	$/t	11,795.8
Payback from first MREC	Years	2.5
Post-tax NPV_8	$ M	283.3
Post-tax NPV_{10}	$ M	207.0
Post-tax IRR	%	25%
Operating margin	%	42%

Note: TREO – Total Rare Earth Oxide; ROM – Run of mine; MREC – Mixed Rare Earth Carbonate; EBITDA – Earnings before interest tax, depreciation and amortisation; opex – operating expenditure.

11.11 Upside scenario

An upside scenario with higher long-term metal prices has been undertaken. No changes in production, opex, capex or discount rates were made to the financial model. The long-term metal prices applied are as follows:

- Praseodymium oxide price of $174,000/t.
- Neodymium oxide price of $183,000/t.
- Terbium oxide price of $2,083,000/t.
- Dysprosium oxide price of $474,000/t.

Total gross revenues of $4,258 million are reported over LOM for the upside scenario; with planned net revenues of $3,704 million.

The NPV of the upside scenario is $409.9 million, based on a real discount rate of 8%. An NPV of $231.3 million is reported using a real discount rate of 12%. A post-tax IRR of 32% and a payback from the construction start date of 4.0h years, and a payback from first TREO production of 2.0 years is reported. An operating cashflow margin of 50% is noted.

12 PROJECT EXECUTION

A high-level planned schedule has been undertaken for the overall Project. Key milestones are highlighted in the Level 1 schedule (Table 12.1). The schedule was based on industry benchmarking, scope of work and a general deliverables list. Snowden Optiro assumes a seamless advancement between the various phases, as the Project advances. The overall schedule is five years to first TREO being produced, which includes 18 months for a PFS, one year for a FS, two years construction and a six-month production ramp-up. Project financing will be applied for, for pre-production funding and Project construction.

Initial major work packages would include:

- Ordering of long-lead items
- Upgrading of the access road to the Project
- Permanent and temporary accommodation
- Progressing of diesel generator yard and grid power infrastructure
- Wellfield development
- Earthworks and civils for all primary terraces and buildings.

Table 12.1 Planned milestones for the Project.

Milestone	Milestone date/ duration
Submission of Mining Concession application	Q4 2023 (Achieved)
Prefeasibility study	18 months to March 2025
Feasibility study	12 months to March 2026
Value engineering, FEED and financing	Nine months to December 2026
EPCM tendering	November 2026
Early works commencement	December 2026
EPCM award	January 2027
Construction commences	Two years to December 2028
First TREO to be produced	December 2028
Production ramp-up	Six months to June 2029
Steady state of 187.5 kt per quarter (750 kt/a)	Q3 2029

Note: FEED – Front end engineering design; EPCM – Engineering, procurement, construction management; TREO – Total Rare Earth Oxide

Source: Snowden Optiro, 2023

An engineering, procurement, construction management (EPCM) execution strategy has been recommended for the Project. The planned contracts would be mixture of lump sums for equipment supply and cost-reimbursable contracts with performance incentives for construction, where required. This execution strategy provides the Company with greater control over the outcomes specifically regarding health, safety, security, environment, and community issues, and more flexibility with respect to timing of the activities, and input into design.

The EPCM provider’s scope of work would cover most of the execution phase activities and will include the services required to design, construct, pre-commission, and commission the Project.

A formal project execution schedule will be undertaken during the PFS and FS. Critical path items and float will be incorporated into the master schedule. Schedule assumptions and cognisance of rainfall periods will be incorporated into the project execution schedule and associated plan.

During the PFS, a detailed work breakdown structure (WBS) template will be undertaken for both capital and operating expenditure.

A procurement operating plan and a long lead equipment list will be formulated during the FS.

Post-FS, a project execution plan and an operational readiness plan will be undertaken.

Detailed risk assessments and hazard identification (HAZID) and hazard and operability (HAZOP) study(ies) will be undertaken during FS. The risk assessments will inform the overall risk management plan.

12.1 Project budget

Table 12.2 below contains the indicative costs for PFS level work at Monte Muambe.

Table 12.2 Indicative budget for PFS

PFS component	Budgeted cost ($)
Operations overheads and salaries	400,000
Drilling; 6,000 m RC and 1,000 m DD	468,000
Assays	302,600
Metallurgical testwork and studies	400,000
Geotechnical studies	70,000
Geotechnical drilling and testwork	30,000
Mining studies	150,000
Mineral resource updates	60,000
Hydrology and hydrogeology studies	100,000
Tailings and waste studies	100,000
Infrastructure	100,000
Equipment	18,500
Contingency	100,000
TOTAL	**2,300,000**

13 CONCLUSIONS AND RECOMMENDATIONS

13.1 Snowden Optiro conclusions

It is expected that Altona will undertake a PFS as the next stage of project development, based on the positive outcome of this CPR and Scoping Study. Specific conclusions are summarised below:

13.1.1 Geology and resources

Not all carbonatites carry REE mineralisation at Monte Muambe. Both low grade (0.5 to 1% TREO) and high grade (>1% TREO) mineralisation is encountered in specific REE-enriched parts of the carbonatite intrusion.

- Low grade mineralisation (LGM) is mostly associated with Ca-carbonatites.
- High grade mineralisation (HGM) is associated with both Ca-carbonatites and Mg-carbonatites, and often extends across boundaries between these two lithologies.

This suggests that lithology is not the sole control for HGM, and that the current geometric distribution of REE at Monte Muambe may be the result of post-magmatic remobilisation and redeposition across geological boundaries. However, two distinct geochemical domains were identified based on their REE and Nb grades (for all drill data across the complex).

- Low grade mineralisation with 0.5% to 1% TREO (average ~0.7% TREO) and a high level of Nb (typically above 500 ppm Nb)
- High grade mineralisation with over 1% TREO (average ~2.4% TREO) and a low level of Nb (typically below 500 ppm Nb).

The REEs at Monte Muambe are contained in three minerals namely bastnaesite (the most common), florencite and synchisite.

Mineral Resources have been estimated for Targets 1 and 4. The Monte Muambe 2023 Mineral Resource has been classified and reported in accordance with the JORC Code (2012). The classification is based on the following parameters:

- Blocks within the pit shells informed with a minimum drill hole spacing approaching 80 m along strike by 80 m across strike, in addition to demonstrated geological and grade continuity, and where the grade has been extrapolated no more than 35 m across strike, was classified as an Indicated Mineral Resource.
- Where informed by wider spaced drilling and/or where geological and grade continuity was assumed, the Mineral Resource was classified as an Inferred Mineral Resource.

Additional drilling, petrological, mineralogical and geochemical work will be necessary to improve understanding of the REE mineralisation controls at Monte Muambe.

The 2023 maiden Mineral Resource appropriately reflects the available data and geological understanding. No material discrepancies were identified with the available data. Two spatially consistent mineralised domains were interpreted identified, at 0.5 to 1.0% and ≥1.0% TREO. Grade variability within these interpreted domains was low. There is reasonable confidence that the estimated grade suitably reflects the available geological understanding and sampling. The Mineral Resource has been reported in accordance with the JORC Code (2012) reporting guidelines and has been constrained by an optimised pit shell to reflect the reasonable prospects for eventual economic extraction. The Mineral Resource employed a range of metrics to reflect the confidence in the estimate.

On-going drilling and geological work is required to continue to develop the spatial geological understanding of the deposit and to improve the local confidence of subsequent Mineral Resources. This includes improved understanding of the deposit geochemistry, weathering/oxidation features as well as collecting additional bulk density samples.

13.1.2 Mining

The mining method is based on conventional open pit using truck and shovel, and drill and blast, coupled to a ROM stockpile. Although the rock is largely classified as weathered, ore and waste rock will require drilling and blasting.

Both ore and waste will be excavated in 5 m flitches following mark-out by grade control. Ore will be hauled to either the ROM pad and tipped onto a designated ore finger or a designated low-grade stockpile. All mine waste will be hauled directly from the pit and placed onto a designated location of the TSF dam wall.

No Ore Reserve was estimated or reported.

13.1.3 Metallurgical Testwork

The existing testwork shows promising potential to upgrade the REE content by conventional beneficiation and hydrometallurgical routes. However this aspect of project development is the highest risk to project economics and overall viability. A detailed definition of process flowsheet and quantitative information on recoveries is required at the next stage of project development.

13.1.4 Infrastructure

Development of a rational site block plan is to be undertaken during the PFS stage, which will incorporate all the services that support the mine, beneficiation plant and support infrastructure within the mine lease area. It is recommended that high resolution topographic survey of the area and survey along approximately 35 km access road upgrade route (and village by-pass routes) suited to infrastructure design work is undertaken in advance of the next study stage.

There are benefits from the topography of the crater where this assists with water storage and gravity flow of waterborne services as well as storm water management, which naturally drains out at the southern rim of the crater. A disadvantage of the crater topography is the impact on the mine access road that negotiates 200 m to 300 m change in elevation from the rim into the basin at a maximum 10% gradient. A 2 km to 3 km long ramp into the crater may require re-thinking the efficacy of the current north-south route.

The construction and ongoing maintenance cost of approximately 35 km access road upgrade and bridges is a significant factor as the mine is reliant upon transport security for delivery of consumables to the mine and export of product from the mine.

Engineering geotechnical investigation work is recommended to evaluate the near surface founding conditions for the beneficiation plant, support infrastructure as well as sourcing construction materials for engineered terrace construction, mine road building and aggregates for concrete manufacture.

Initial indications are that a competent founding horizon is within 2 m of the natural ground surface, which substantially reduces the cost and risk of civil works construction, especially for the beneficiation plant.

Harvesting competent waste rock at shallow depth from the future mining areas for road and terrace construction and for concrete manufacture may be possible during the early works infrastructure build programme and prior to TSF wall construction.

Development of the manpower histogram to greater detail in the PFS will improve the accuracy of support infrastructure design and cost of construction for services (water and sewerage reticulation, solid waste disposal, etc), accommodation and non-process infrastructure (offices, camp, kitchens and change house, etc.).

13.1.5 Tailings and waste

As most of the design and costing for the TSF is based on high-level estimates and experience, further engineering and costing is required going forward in the PFS.

The waste rock material from the mining activities planned for the Target 1 and Target 4 pits will be deposited on designated WRDs used in the construction of the TSF containment walls. The WRDs are situated in close proximity to the pits to reduce haulage distances. The WRDs will be developed over a total footprint of approximately 40 ha and to a final height of approximately 50 m. Detailed modelling to be undertaken to confirm TSF waste rock requirement and additional capacity required for the waste rock dumps.

13.1.6 Project economics

Preliminary work has provided a positive cashflow model for the project. At the moment this is defined with a precision of +35% / -30%, and the next step will involve detailed input from all sections of the project.

13.2 PFS work plan recommendations

A number of studies need to be undertaken prior to project execution. During the PFS and FS, all items discussed in Section 12 (master schedule, project execution plan, HAZID, HAZOP, WBS), will be progressed. Specific items that should be addressed are summarised in relevant sections below.

13.2.1 Exploration

- Use the improved mineralisation model to attempt identifying new targets, including blind targets.
- Continue improving mineralisation model through mapping as well as academic research.
- Exploration drilling at T3, T9, T11, and any other potential high-grade target
- MRE update
- Data centralisation

The resource update should cover tonnage increase, as well as improve the level of confidence within the pits to Measured and Indicated. Resource drilling and studies should include:

- Infill drilling at T1 and T4; 2 to 4 sections on a spacing of 20 m along strike x 20 m across strike and outside of that a broader pattern of 40 m x 40 m throughout the rest of the T1 main mineralisation focusing on high grade domain.
- Down-dip drilling at T1 and to 150 m below surface at T4.
- Re-evaluation of T6 through with a pit optimisation exercise to see if this should be considered in the MRE and if further drilling then required.
- Resource drilling on any target confirmed through exploration drilling.
- Geotechnical drilling (triple tube DD) in the two open pits. This must be structurally orientated and structurally logged and geotechnically logged to provide input for future pit optimisations.
- More spatially distributed density data from the planned diamond drilling. Snowden Optiro recommends density sampling is done every assayed interval.

13.2.2 Geometallurgy / processing

Geometallurgy and process flowsheet design will be a priority activity during the PFS. The Scoping Study sighter testwork forming part of this Scoping Study provides a preliminary assessment based on a possible flowsheet.

Future testwork is expected to follow the flowsheet progressively, with upstream sections being developed to produce representative test material for downstream sections. Note that when referring here to a flowsheet, it does not imply the concept study flowsheet necessarily, as alternative processing routes could be selected as the sighter testwork progresses and associated technical and cost-benefit assessments are implemented. The beneficiation and hydrometallurgical plants should be viewed from a functional perspective, comprising the following main steps of progressive processing:

- Mineralogical and geo-metallurgical assessment.
- Beneficiation flowsheet development:
  - Optimal grind size
  - Crushing/ grinding parameters including work and abrasion indices

  - Need for desliming
  - Requirement for multi-stage flotation
  - Thickening and filtering characterisation testwork.
- Hydrometallurgical flowsheet:
  - Optimise gangue leach conditions
  - Evaluate rare earth leach options (hard HCl leach and/or cracking)
  - Reagent selection for purification via pH adjustment (lime or caustic soda)
  - IX removal of uranium
  - Mixed rare earth carbonate precipitation
  - Thickening and filtering characterisation testwork
  - Radionuclide deportment study.

Further detailed metallurgical studies for Monte Muambe are underway and currently focused on advanced metallurgical testwork. A 70 kg representative ore sample is with Auralia Metallurgy in Perth (Australia), and another 100kg ore sample has been received by SGS Lakefields in Canada. The sample at SGS Lakefields will first undergo extensive feed characterisation including Electron Microprobe Analysis and TIMA-X analysis. TIMA-X is designed to provide quantitative mineral speciation and distribution, as well as characterisation, grain size attributes, degree of liberation and associations of minerals of interest.

Following feed characterisation, test work will focus on producing a high-grade Rare Earth concentrate in order to improve the economics of the Mixed Rare Earth Carbonate production process. The possible separation and recovery of fluorspar, another critical raw material present in the ore at Monte Muambe, will also be assessed. Flotation test results are expected in Q2 2024.

Geo-metallurgical modelling should be implemented, looking in particular at optimising the cut-off grade and at decisions as to what is ore and what is waste (possible including NdPrO/ TREO ratio modelling). This will be necessary to domain the geological and resource model, allowing more efficacious mine planning, scheduling, and process planning.

During the PFS, work will be undertaken to define the proportion of oxide, transition and fresh material reporting to the plant on a quarterly/annual basis over LOM. There is likely to be some effect on processing (flotation) from different degrees of weathering; however, with the metallurgical testwork that has been done it cannot be accurately defined. The sulphide content of the ore will also need to be quantified during the ongoing testwork.

On a marketing/ sales consideration, further downstream processing from MREC product through part separation of a NdPrO product. There is a possibility of a regional hydrometallurgical plant facility to be considered, involving cooperation between different REE mining operations in the Southern Africa region.

Separation of the 15 Rare Earths present at Monte Muambe from their ore, with a focus on neodymium, praseodymium, terbium and dysprosium, is a complex process. Metallurgy is a critical component of rare earths projects development. Beside process design and costing, key outputs will also include products specifications to enable discussions with potential off-takers.

13.2.3 Mining geotechnical

The mining geotechnical studies should form part of an initial work program and comprise:

- Defining a geotechnical drilling program for the PFS (the drilling will be managed by Altona's exploration team). The outcome of the initial pit optimisation studies will provide guidance on the number and location of diamond drill holes required for the geotechnical studies.
- Onsite geotechnical logging of drill core and data collection. This is to ensure that logging and sample collection are done of the required quality and standard. The site visit will also entail discussion with exploration geologists to ensure all major structures within the vicinity of the deposits have been mapped.

- Selection of drill core for off-site testing, selection, and management of the geotechnical testing laboratories. It is expected that drill core will be transported to recognised laboratories in Johannesburg for rock testing (likely tests will include, rock – UCS, tensile strength, saw cut shear strength, soil – Atterbergs limits, PSD, triaxial tests).
- Structural interpretation – All available drilling and mapping data will be compiled into a geotechnical database and analysed to identify major structures that will have significant impact on wall stability.
- Assessment of data, development of geotechnical parameters and completion of various slope stability assessments. Rock mass classifications will be calculated from the geotechnical drilling data. The rock mass classification and structure data will be processed and compiled to provide an understanding of the deposits and their surrounds, comprising a series of spatial domains of each delineating material with distinct geotechnical properties and behavioural characteristics.
- Slope stability analysis. Rock mass and structurally controlled failure analysis for each wall of the open pit (overall and batter scale), particularly critical sections identified during the studies targeting agreed factor of safety. Slope stability analyses will be undertaken with kinematic and/ or limit equilibrium and numerical methods.
- Developing geotechnical design parameters for the open pit(s). Design parameters for mine planning optimisation purposes by pit sectors in a form of polygon or solids indicating the boundaries of each slope design recommendations.
- Excavatability, trafficability and blastability assessments, defining potential areas of free dig, poor road conditions requiring increased pavement/sheeting thickness and indicative blast patterns and power factors for cost evaluation purposes.

13.2.4 Mine planning and Ore Reserve

Initial work program

To initiate early works and the gain an understanding of the likely scale of operations, the initial mining studies will include:

- Preliminary pit optimisation and schedule to guide to provide guidance to Altona and consultants (geotechnical, hydrology, geohydrology, metallurgical testwork, processing, environmental).
- Scenario analysis to identify best throughput rate and size of resource required – Identify the best project development plan over the entire life of Project to be identified including the scale and quality of additional resources required to achieve the PFS ROM target.
- Mining cost assessment – based on initial schedules, develop contract mining cost.
- A site visit by the Ore Reserve Competent Person.

Detailed prefeasibility studies

Once adequate metallurgical testwork has been completed to reliably inform recovery and process cost parameters, the following work can be completed:

- Open pit optimisation: Complete a pit optimisation using industry optimisation software to determine the economic limits of the pits for design. For the Ore Reserve case, this would be completed for Measured and/ or Indicated Resources only. Sensitivity analysis should be completed to justify the selected pit, including optimisation with inferred resources.
- Incorporate geo-metallurgy modelling and individual element grade/ recovery and pricing in optimisation studies
- Pit design: Pit designs from shells derived from pit optimisation, including any stages in a generalised mining package using design criteria provided by the geotechnical work. Pit staging to identify opportunities to minimise the amount of waste mined in the early years. These studies will need to consider the amount of waste required to progressively construct the TSF.

- Waste dump and stockpile design: Simple waste dump design for each pit in a generalised mining package. More detailed studies may be required to incorporate the waste dump into the TSF.
- Major haulage roads: Complete haul road layouts connecting the pits to the waste dump, ROM pad and stockpiles.
- Mining schedule: Develop quarterly mining schedules of ore and waste movements, and stockpile movements, by bench and, if necessary, generate a processing schedule.
- Develop capital and operating costs for mining of ore and waste and ROM loading. Grade control and owner's mining costs should also be developed.
- The Ore Reserve Competent Person for the mining aspects of the Ore Reserves will need to complete Section 4 of Table 1 (JORC Code, 2012) for the Ore Reserve in conjunction with other Competent Persons.
- Identify opportunities to stockpile LG mineralisation for processing at the end of the mine life.

13.2.5 Environmental studies

- Start baseline studies as soon as possible.
- Environmental compliance as part of the Mining Concession application (EMP, ESIA). ESG planning to World Bank level.
- As part of the PFS, see how the carbon footprint of the proposed product can be minimised through locally available opportunities.

13.2.6 Infrastructure studies

- Logistics optimisation – road vs different rail options.
- Consider whether all or part of the processing should be undertaken at a port as opposed to on site.
- Power sources mix optimisation (capex, opex and carbon footprint).

13.2.7 Tailings/ waste management

Tailings

As there are several mineral targets surrounding the infrastructure area, one of the key focus areas will be sterilisation drilling, to determine if there are any mineralised zones below the 60 ha footprint required for the TSF.

The PFS will require a detailed site selection and associated surface geotechnical investigations. A key requirement will be to conduct geochemical static and kinetic leach testing on the types of ore/ tailings/ waste to determine the future design/ lining of the TSF.

The planned PFS will identify several options and determine the best site or sites for tailings disposal. Optimization of facility locations on site map showing the topography; simple general arrangement drawings of major equipment items.

Currently there are good quality topographic maps; but a soil conditions report for the foundation determinations and more accurate basic preliminary quantities will be required.

Waste rock

The following recommendations are made for inclusion in the scope for the prefeasibility phase:

- Hydrology – This study should include regional climate, catchment characteristics, surface flow patterns and quantities.
- Geohydrology – A geohydrological study is recommended to accurately determine the depth to and extent of groundwater resources / aquifers and the characteristics thereof.
- Site selection – A more detailed site selection study with consideration to technical, environmental, and social factors.
- Geotechnical investigation – A geotechnical investigation may be undertaken for the selected WRD sites and general area to provide an initial indication of the near surface soil- and groundwater conditions and identify any problematic subsoil conditions.
- Geochemistry – Geochemical analysis and characterisation of the waste rock is required to confirm the assumptions made regarding the non- acid generating potential and high alkalinity of contact water.
- Design criteria – Confirm and update design criteria as required.
- Cost estimates – Improvement on capital and operational cost estimates.

13.2.8 Marketing

As part of the PFS, join a Responsible Sourcing organisation and integrate Responsible Sourcing processes. Develop marketing side of business as part of PFS. Offtakes and integration with ROW supply chains (existing and projects) to be addressed in PFS.

13.2.9 Project economics

Relevant studies need to be undertaken to improve granularity and accuracy of the opex and capex estimates, production and planned recoveries.

Appendix A
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Appendix B

Table 1

Section 1 Sampling techniques and data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code (2012) explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• The database compiled by Altona Rare Earths Plc (Altona) for the Monte Muambe rare earth project (the Project or Monte Muambe) contains 113 holes totalling over 7,800 m and comprises of diamond drilling (DD) and reverse circulation (RC) holes. The database does not include twinned RC-DD holes. • The drill hole dataset directly used in the Mineral Resource estimation comprises a total of 2 DD holes (299 m) and 121 RC holes (9,413 m). • All extracted DD core was sampled and assayed. Core sampling was done over a nominal length of 1 m with variations taking into consideration lithological contacts. Individual sample lengths varied from 4cm to 336cm with an average of 62cm. Quarter cores (PQ and HQ) and half cores (NQ) were submitted to Intertek Genalysis laboratory for chemical analysis. Samples not competent enough to be split with the core saw were bagged, homogenized, and split using a riffle splitter. • Samples from RC drilling were collected at 1 m intervals. All RC samples from the 2021 drilling campaign were composited at 3 m intervals. Samples selected on the basis of their REE content (TREO>0.5% for 3 m composite results) were re-composited at 2 m intervals and resubmitted to the laboratory. Samples from the 2022 RC drilling campaign selected on the basis of preliminary onsite pXRF assay results (TREO>0.5%) were composited over 2 m intervals and submitted to the laboratory. In both cases the weight of the submitted splits was about 3 kg. • Sampling was carried out in accordance with Altona SOPs which are in line with industry practice. During RC drilling the drill string and cyclone were regularly flushed with air and cleaned. 1 m cutting bags weights were monitored and recorded. RC cuttings were split using a 3-tier riffle splitter which was cleaned between samples. • All samples were despatched by road to the Intertek Genalysis facility in Boksburg, South Africa for sample preparation, with the pulps forwarded by airfreight to Intertek Genalysis Perth laboratory for assay. DD samples were dried, crushed to ~10 mm, split and

Criteria	JORC Code (2012) explanation	Commentary
		pulverized to 85% or better passing 75 micron. RC samples were dried, split and pulverized to 85% or better passing 75 micron. • Samples were assayed for Al, Ba, Ca, Ce, Cr, Cs, Dy, Er, Eu, F, Fe, Ga, Gd, Hf, Ho, K, La, Lu, Mg, Mn, Na, Nb, Nd, P, Pr, Rb, S, Sc, Si, Sm, Sn, Sr, Ta, Tb, Th, Ti, Tm, U, V, W, Y, Yb, Zr, LOI. • A total of 2,960 samples including DD samples, 3 m composites and 2 m composites were sent to the laboratory and assayed. • An overall total of 6,733 assays were used for the Mineral Resource estimation.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• The diamond drilling rig was a trailer mounted Atlas Copco CS14. Diamond drill (DD) holes were started in PQ diameter, with the diameter reduced to HQ and if necessary NQ as dictated by ground conditions. 15.5% of diamond drilling was done in PQ diameter, 63.9% in HQ and 20.6% in NQ. Because of the disseminated nature of the mineralization, it was not considered necessary to do core orientation. • The RC rigs were a truck mounted Smith Capital 14R6H with a 21bar compressor (2021 and 2023 drilling campaigns) and a track mounted Hanjin Power7000SD (2022 drilling campaign). The RC bit has a 4 ½ diameter.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• The DD cores were checked against the driller's core blocks and recovery was recorded. The presence of cavities was recorded based on information provided by the driller and observations on the core. • DD core recoveries varied from 17% to 100%, with an average of 83%. Short runs were used to maximize sample recovery when necessary. • RC recoveries were systematically monitored and recorded, and regularly assessed. The presence of cavities was recorded based on information provided by the driller. 1 m samples were weighted and their moisture content recorded.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or*	• The entire length of each hole was logged by trained geologists. Lithology, mineralogy, colour, weathering, grain size, texture, fabric and alterations were logged. • RC samples pXRF logging was done on-site using a Hitachi X-MET8000 device with a 50 kv anode designed to assay Ce, La, Nd,

Criteria	JORC Code (2012) explanation	Commentary
	costean, channel, etc) photography. • *The total length and percentage of the relevant intersections logged.*	Pr and Y. For RC samples, a 50 g sub-sample was split from each 1 m sample using a 1-tier riffle splitter. Each sub-sample was split further and placed in an XRF capsule for assay. The pXRF was set up in bench top mode. Preparation and assay were done in standard conditions. The sum of the 5 above-mentioned elements was calculated as oxide %. Orientation, QAQC and comparisons with laboratory results show that this sum provides a reliable proxy of the actual TREO%. Accordingly, pXRF logging results were used to guide the day-to-day implementation of the drilling program and to select mineralized samples (TREO>0.5%) to be sent to the laboratory for assay. pXRF assay results were not used for the Mineral Resource estimation. • Lithology determinations on RC chips were supported by the preliminary pXRF assays done on site, with SiO_2 being used to distinguish fenite from carbonatite and from mixed lithologies, and MgO to distinguish two geochemically different suites of carbonatites. • Gamma spectrometer logging was carried out using a hand-held gamma spectrometer to do one reading in each RC cutting bag (RC samples) and to do spot readings at 50cm intervals of cores (DD samples). • No geophysical tools were used to determine element grades. • Geology logging was qualitative and pXRF logging was quantitative. • All DD core trays, and RC chip trays were photographed in standard conditions and white-balanced.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• For PQ and HQ diameters, a ¼ core was submitted to the laboratory. For NQ diameter, ½ cores were submitted to the laboratory. The cores were cut on site by a trained technician using a core saw with a diamond impregnated blade. Cores that were not competent enough to be split with the core saw were bagged and split using a riffle splitter. • 1 m RC samples, as well as 3 m and 2 m RC composites were split using a 3-tier riffle splitter. When necessary, samples were sun-dried in a protected environment before splitting. • Samples were prepared at Intertek Genalysis's Boksburg (South Africa) facility. DD samples were dried, crushed to ~10 mm, split and pulverized to 85% or better passing 75 micron. RC samples were dried, split and pulverized to 85% or better passing 75 micron.

Criteria	JORC Code (2012) explanation	Commentary
		• Field duplicates were prepared by submitting another ¼ core or ½ core (for DD core samples) or by preparing a 2nd composite from the same interval using the same splitting and compositing method (RC samples). • Field duplicates, OREAS certified reference materials (CRMs) and blanks (locally procured quartzite) were inserted in sample batches at fixed intervals, with a frequency of 1 of each type in 27 samples which equated to insertion rates of 3.7% (Duplicates), 3.6% (Blanks) and 3.5% (CRMs) • 112 pulp samples were submitted to Nagrom laboratory in Perth for secondary (umpire) analyses. This equates to 5.9% of the original pulp samples analysed at Intertek Genalysis. • Both Intertek Genalysis and Nagrom laboratories also conducted and reported internal QAQC checks. • QAQC data is stored in the Project database. • A review of QAQC data did not highlight any significant sampling or analytical issues. The QC sample insertion rate is slightly low, but is being increased during the next stages of project development.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• Sample analysis was carried out by Intertek Genalysis' Perth laboratory an accredited status (ISO/IEC 17025). • All samples were assayed for Al, Ba, Ca, Ce, Cr, Cs, Dy, Er, Eu, F, Fe, Ga, Gd, Hf, Ho, K, La, Lu, Mg, Mn, Na, Nb, Nd, P, Pr, Rb, S, Sc, Si, Sm, Sn, Sr, Ta, Tb, Th, Ti, Tm, U, V, W, Y, Yb, Zr, LOI. • Major elements and some trace elements (including Ce and La) were assayed by Li Borate fusion followed by ICP-OES. • Trace elements (including all REE, U, Th and Nb) were assayed by Li Borate Fusion followed by ICP-MS. • Fluoride was assayed by alkaline fusion in a nickel crucible followed by specific ion electrode (SIE) analysis. • Field duplicates, OREAS certified reference materials (CRMs) and blanks (locally procured quartzite) were inserted in sample batches at fixed intervals, with an approximate frequency of 1 of each type in 27 samples. Actual insertion rates were 3.5% CRMs, 3.6% Blanks and 3.7% Field Duplicates. • The laboratory also conducted and reported its own QAQC checks. • Pulps for 1 in 17 (5.9%) samples for DD and all RC 2 m composites

Criteria	JORC Code (2012) explanation	Commentary
		were sent to Nagrom laboratory for external (umpire) assays. Where available, 3 m composites from the 2021 drilling campaign were compared with 2 m composites. For Ce and La, ICP-OES and ICP-MS results were compared. The QAQC review results indicated acceptable levels of accuracy and precision and did not highlight any significant assaying issues.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Significant intersections were verified by Altona's CEO. • No twin DD-RC holes were drilled. • Field data was transferred into Microsoft Excel sheets stored in a Dropbox folder backed-up on an off-site server in real-time. Data entry was done by trained geologists, under the responsibility of the Project Manager. The database was regularly backed up on a second Dropbox account, and on the Company's Sharepoint backup system. Scans of all paper documents (driller's daily reports, logs etc) are stored digitally in the database. • Digital data were checked and validated against the original field sheets. • REE, U and Th are reported by the laboratory as element ppm. Grades reported in the Mineral Resource estimation as TREO% or as NdPrO% involved conversions from element ppm to oxide %. The following conversion factors were used: ○ La to La_2O_3 – 1.1728 ○ Ce to CeO_2 – 1.2284 ○ Pr to Pr_6O_{11} – 1.2082 ○ Nd to Nd_2O_3 – 1.1664 ○ Sm to Sm_2O_3 – 1.1596 ○ Eu to Eu_2O_3 – 1.1596 ○ Gd to Gd_2O_3 – 1.1526 ○ Tb to Tb_4O_7 – 1.1762 ○ Dy to Dy_2O_3 – 1.1477 ○ Ho to Ho_2O_3 – 1.1455 ○ Er to Er_2O_3 – 1.1435 ○ Tm to Tm_2O_3 – 1.1421 ○ Yb to Yb_2O_3 – 1.1387 ○ Lu to Lu_2O_3 – 1.1371 ○ Y to Y_2O_3 – 1.2699

Criteria	JORC Code (2012) explanation	Commentary
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• The project's coordinate system is UTM Zone 36s, WGS84 Datum. All reported data uses this coordinate system. The position of the project's reference point is not tied to the local grid and its position was acquired by the RTK dGPS's system base station. • Hole collar positions used for the Mineral Resources estimate were georeferenced using a Kolida 20S RTK dGPS system, owned by Altona and operated by a trained technician. • Except for 4 DD holes which were drilled vertically, all holes were drilled at a dip angle of 55 degrees. The mast angle was checked using a specially designed spirit level. • Down-hole surveying was done in plastic tubing inserted in holes after the drill string was removed. Not all holes could be surveyed due to collapses and blockages. 80% of the DD holes and 59% of the RC holes having a depth >30 m were surveyed using an EZTRAC system, at 6 m depth intervals. • The Altona drill hole used RTK dGPS survey locations, with the exception of fourteen drill holes. These holes were projected onto the available DEM. All drill hole collars were then reviewed spatially against available DEMs and no discrepancies identified. • Digital Elevation Models (DEM) were prepared by Snowden Optiro for Target 1 and Target 4 (0.921 and 1.357 km^2 respectively) using drone photogrammetry data collected by Altona with a non-RTK drone. Photogrammetry data processing included the use of RTK dGPS-surveyed Ground Control Points (GCPs). The DEM has a nominal resolution of 2 to 3 metres. • The Competent Person is of the opinion that the accuracy and precision of the survey data is suitable for resource estimation.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• DD and RC drilling conducted in 2021 was aimed at testing potential targets and did not follow a regular pattern. • RC drilling conducted in 2022 at Target 1 was done along fences with most holes having an azimuth of 213 degrees and a dip angle of 55 degrees. The distance between fences ranged from 70 to 80 m and the distance between holes of the same fence ranged from 50 to 75 m. • RC drilling conducted in 2022 at Target 4 was done along two fences 150 m apart, with a spacing between holes of about 65 m. • RC samples used in the Mineral Resource estimation were 2 m

Criteria	JORC Code (2012) explanation	Commentary
		composites. The choice of this composite length was based on information from preliminary modelling done in the context of an Exploration Target estimate prepared by Rock and Stock Investment on 19 July 2022.

Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*
		• Two geochemically distinct types of mineralization have been identified at Monte Muambe:
		  ◦ Low-grade mineralization: TREO% between 0.5 and 1% (average 0.73%), and an average NdPrOxide/TREO ratio of 0.18.
		  ◦ High-grade mineralization: TREO% above 1% (average 2.38%), and an average NdPrOxide/TREO ratio of 0.14.
		• High grade mineralization at Target 1 forms a relatively well-defined body trending in a WNW-ESE direction and dipping towards the NNE at an angle of 40 to 50 degrees. Its length is about 500 m, and its thickness varies from 30 to 70 m, with inter-burden intervals of 5 to 20 m encountered in some holes. Drill holes intersected the mineralized zone at approximately 80 degrees and borehole orientation was appropriate for this geometry.
		• High grade mineralization at Target 4 forms a sub-vertical pipe about 150 m in diameter. Drill holes intersected the envelope of the mineralized zone at a +/- 35 degrees angle and their orientation was suitable for this geometry.
		• No significant sample bias is considered to have been introduced by the drilling orientation and its relation to the mineralization geometry.

Sample security	• *The measures taken to ensure sample security.*	• Sample security is managed by the Altona Project Manager. RC samples awaiting shipment are stored in plastic woven sacks in an open shed. DD core trays are stored in a locked section of the core shed. Batches of samples are checked by a technician from the Tete Provincial, Infrastructure Service, Department of Mineral Resources and Energy and collected by the transporter on site. Samples are transported in a locked truck to the sample preparation laboratory in South Africa. Pulps are air freighted by the laboratory to its Perth facility for assay. • The laboratory audits the samples on arrival and reports any discrepancies with the submission documentation to the Company.

Criteria	JORC Code (2012) explanation	Commentary
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Snowden-Optiro has conducted a review of primary and QAQC data. This included a site visit by Mr Robert Barnett on behalf of the Competent Person Andrew Scogings from 7 – 10 August 2023. • The Competent Person is of the opinion that the sampling techniques are suitable for the type of mineralization. The Competent Person is of the opinion that the sample database is adequate for the 2023 Mineral Resource estimated. Altona has advised that an improved database system will be implemented going forward. • There has been no other formal audit of the project or data at this stage.

Section 2 Reporting of exploration results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code (2012) explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The Project is held under Prospecting Licence (*Licença de Prospecção e Pesquisa*) LPP7573L (the Licence), issued in accordance with the Mining Law 2014. The Licence covers a surface area of 3,939.87 Ha (39.3987 km^2), and is valid for Fluorspar, REEs and Associated Minerals. The Licence was granted on 22 May 2017 to Ussokoti Investimentos Sociedade Unipessoal for an initial five year term. On 26 October 2022, the Licence was transferred to Monte Muambe Mining Limitada, and renewed for a final three year term, i.e. up to 22 May 2025. • The Licence is in good standing and there are no known impediments to its validity or to the Company's ability to secure a Mining Concession. • Altona's interest in the Licence is through a Farm Out Agreement dated 23 June 2021 between Ussokoti Investimentos (the original owner of the licence – see section 3.4 of this report), Altona, Monte Muambe Mining Lda (MMML) and its original shareholders. The Farm Out Agreement gives Altona the right to earn up to 70% of MMML in a phased manner, subject to the completion of certain conditions and milestones. Each transfer of shares requires the approval of the Minister, Ministry of Mineral Resources and Energy (MIREME).

Criteria	JORC Code (2012) explanation	Commentary
		• Beside exploration rights, the Licence gives the licencee a preferential right to an application for a Mining Concession. The Company plans to lodge a Mining Concession application in Q3 2023. Mining Concessions have a validity of up to 25 years, renewable once for an equal period. Mining activities on a Mining Concession also require the obtention of Land Rights (Direito de Uso e Aproveitamento da Terra – DUAT), and an Environmental Impact Study for Category A activities.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Previous exploration done at the Project site includes fluorspar exploration by Grupo Madal (1998), and fluorspar and rare earths exploration done by Globe Metals & Mining between 2009 and 2012.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Monte Muambe deposit is a ~3.5 km diameter weathered sub-volcanic carbonatite intrusion. The intrusion occupies the inner part of a circular structure resulting from the differential erosion of indurated Karoo sandstones. A fenite alteration zone lines the contact between the carbonatite and the sandstones. At a detailed level, the geometric relations between the carbonatites and the fenites are complicated by faulting. Pyroclastic deposits from the former volcanic edifice are preserved in parts of the structure. • REE mineralization, and high-grade mineralization in particular, are heterogeneously distributed in the intrusion. High-grade mineralization forms relatively well-defined bodies of varying shape, size and orientation. • Fluorspar is known at Monte Muambe and was previously explored. It occurs as disseminations in carbonatite, and as botryoidal veins cutting across the carbonatite. An Inferred fluorspar Mineral Resource totalling 1.63 mt at 19% fluorite was defined by Globe Metals & Mining in 2012. Altona considers the size and grade of the fluorspar occurrence, with little potential for increase, are too low to be of economic interest. • REE-bearing minerals (mainly bastnaesite, but also florencite and synchisite) occur disseminated in the carbonatite or concentrated along mm thin veins and cracks of the rocks. Mg and Fe rich carbonatite tends to have a higher REE content. • The Monte Muambe carbonatites also contain fluorite, apatite, barite, Nb-bearing pyrochlore as well as Fe- and Mn- bearing minerals. None

Criteria	JORC Code (2012) explanation	Commentary
		of these minerals / elements is in sufficient concentration to be recovered. No significant correlations were observed between REE and other chemical elements.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ○ *easting and northing of the drill hole collar* ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ○ *dip and azimuth of the hole* ○ *down hole length and interception depth* ○ *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	• Not relevant – exploration results are not being reported.
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Not relevant – exploration results are not being reported. • No metal equivalents are used.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• High grade mineralization at Target 1 forms a relatively well-defined body trending in a WNW-ESE direction and dips towards the NNE at an angle of 40 to 50 degrees. Its length is about 500 m, and its thickness varies from 30 to 70 m, with inter-burden intervals of 5 to 20 m encountered in some holes. Drill holes intersected the mineralized zone at a 80 degrees angle and their orientation was appropriate for this geometry. Intercepts length is therefore close to the true width of the mineralized zone at Target 1. • High grade mineralization at Target 4 forms a sub-vertical pipe about 150 m in diameter. Drill holes intersected the envelope of the mineralized zone at a +/- 35 degrees angle and their orientation was suitable for this geometry. Intercepts length do not represent the true

Criteria	JORC Code (2012) explanation	Commentary
		width of the mineralized zone at Target 4.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Appropriate plans and sections have been created, but are not included in this statement.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Not relevant – exploration results are not being reported.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Data from the 1998 legacy helicopter-borne geophysical survey was available. This included Total Magnetic Intensity, Analytical Signal, 1st Order Vertical Derivative, Total Count, Potassium, Thorium, Uranium grids. • Altona carried out ground gamma radiometer surveys at Target 1, Target 4 and Target 10. • Altona carried out a comprehensive soil sampling survey covering the carbonatite and fenite outcrop zones, involving the collection of 2,146 samples. Samples were sieved to -500 micron, split using a 1-tied splitter, prepared in XRF assay capsules, and assayed by pXRF on site. The survey was done along a 100 x 100 m grid, with 50 x 50 m and 25 x 25 m infill sampling where necessary. • In August 2023, Altona carried out shallow trenching aimed at exposing substratum outcrops for in-situ pXRF assays at Target 3, 4, and on other prospects at Monte Muambe. Actual assay points were georeferenced using the RTK dGPS system. The objective of this data collection was to determine the position of the envelope of high-grade mineralization zones at surface level. • 20 DD and RC samples selected to represent the lithology and geochemistry variability of Monte Muambe rocks were analyzed by XRD at Intertek Genalysis laboratories to produce semi-quantitative mineralogical data.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• In-fill RC drilling is planned at Target 1 and Target 4 with the aim of supporting future Mineral Resource estimates, and Ore Reserve estimates. Mineralization remains open at depth at both targets. • Additional exploration work is planned at other targets identified through soil sampling, including RC drilling and trenching.

Criteria	JORC Code (2012) explanation	Commentary
		• About 10 tonnes of representative samples from high-grade zones at Target 1 and Target 4 have been set aside for future use in PFS-level metallurgical testing.

Section 3 Estimation and reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code (2012) explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	Random cross-checks were conducted between the supplied original source sheets for the logging, sampling and assay data, and the supplied digital data. The TO value for the last logged interval of eleven holes had <0.5 m rounding discrepancies. These were re-set reset to the correct values prior to desurveying. On compilation and import of the data, random spot checks were completed to confirm the data integrity. The data was then desurveyed in three dimensional space and then reviewed spatially, with no inconsistencies identified.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• Mr Robert Barnett visited Monte Muambe on behalf of the Competent Person Andrew Scogings from 7 – 10 August 2023. Drilling and sampling procedures were observed and density checks conducted. The geological and sampling data are deemed fit for Mineral Resource Estimation.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The current geological understanding of the Monte Muambe mineralisation is still developing. The mineralisation is hosted by a carbonatite, with variable development of fenite as well as mixed carbonatite-fenite rock types, which is variably mineralised. Rare earth mineralisation has been impacted by weathering, with rare earth grades increasing with increased weathering. • Interpretations were prepared using Leapfrog Geo software. Development of the interpretation was primarily based on the TREO grades. Review of the TREO grade distribution, the spatial distribution of grade and observed continuity resulted in selecting a 0.5% on-set of mineralisation, and a 1.0% TREO mineralised grade

Criteria	JORC Code (2012) explanation	Commentary
		cut-off. • Holes informed by laboratory assay and handheld XRF assays were used to inform the geological interpretations. Only laboratory assays were used to inform the estimate. • The 0.5% TREO domain geometry appear as dispersion haloes around the higher grade mineralisation. • Only the T1_11 and T4_11 mineralised domain meet the criteria of a Mineral Resource. • The RC and diamond drill hole samples are assumed to be representative of the material being sampled. • The potential for alternative interpretations on a global scale is considered low to moderate. However, with increased drilling and geological understanding, refinements to the mineralisation is expected. • For estimation, the mineralisation boundaries were treated as hard boundaries, whilst the weathering boundaries were not used for estimation. Both approaches supported by contact analysis. • A broad correlation was observed between magnesium-oxide (MgO) rich carbonatite rocks and the rare earth mineralisation. However, near surface the MgO grades were impacted by the weathering (MgO depletion). There is greater grade continuity associated with the MgO rich rock types and increased greater grade continuity with increased weathering.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• At target 1, the interpreted mineralised is broadly lenticular in shape, strikes approximately 310° with a variable dip ranging from 25 to 35°, dipping towards the northeast. The mineralisation has a strike length approximately 760 m, outcrops on surface, and narrows with depth, pinching out approximately 135 m vertically below surface, The average horizontal width is 42 m. • At target 4, the interpreted mineralised is flat lying, broadly lenticular in shape, and striking between 295 and 320°. The dip that ranges from 05 to 30° to the northeast. The mineralisation has a strike length of approximately 260 m, is 230 m across strike with an average vertical thickness of 46 m. The mineralisation outcrops on surface and extends 96 m vertically below surface.

Criteria	JORC Code (2012) explanation	Commentary

Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*
	• *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.*
	• *The assumptions made regarding recovery of by-products.*
	• *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).*
	• *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*
	• *Any assumptions behind modelling of selective mining units.*
	• *Any assumptions about correlation between variables.*
	• *Description of how the geological interpretation was used to control the resource estimates.*
	• *Discussion of basis for using or not using grade cutting or capping.*
	• *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*
		• Geological modelling was undertaken with Leapfrog Geo (v2022.1), statistical analysis and variogram modelling was completed using Snowden Supervisor (v8.15) and grade estimation was undertaken using Datamine Studio RM Pro (v1.12.113.0).
		• A total of thirteen domain-composite samples combinations had top-cut applied to minimize the local impact of extreme values.
		• As a function of the low variability and coefficient of variation (CV) and relatively low coefficient of skew, ordinary kriging of 2. 0 m top-cut composites were used for estimation.
		• As a maiden Mineral Resource estimate there are no previous estimates for comparison, nor is there any production data.
		• An inverse-distance cubed (ID^3) test estimate using identical estimation parameters but implementing an octant search was completed, with the average absolute relative difference between the two estimation methods of 2.0% at Target 1, and 1.1% at Target 4.
		• Deleterious elements and non-grade variables.
		• Currently there are no selective mining unit assumptions and no no by-product recovery assumed.
		• Independent variogram models were prepared for all variables, on a mineralised domain basis.
		• To maintain the cross-correlation between variables within an estimation domain, identical search parameters were used for all variables within the domain. Estimation employed a three-pass search strategy for all estimates:
		  ◦ At target 1, a disc shaped search with identical along strike and down-dip search distances was used, with a reduced distances perpendicular to the dip plane. The primary search was 150 m x 150 m x 15 m in the plane of the mineralisation, with between 8 and 24 samples. The primary search distance was doubled for pass 2, using the same number of samples. Pass three had the primary distance tripled, with the number of informing samples reduced to between 4 to 12. No restriction on the number of informing samples was used.
		  ◦ At target 4, an ellipse shaped search was used. The primary search was 150 m x 100 m x 80 m in the plane of the mineralisation, with between 8 and 20 samples. The primary

Criteria	JORC Code (2012) explanation	Commentary
		search distance was doubled for pass 2, using the same number of samples. For pass three, the primary distance was tripled, and the number of informing samples reduced to 8 to 12. No restriction on the number of informing samples was used. • Parent block estimation was used for all domains, with discretization of 3 (X) x 3 (Y) x 2 (Z). • The estimate was validated by initially reviewing the composite and estimated grades in plan and section. This was followed by whole of domain comparisons between the naïve and declustered composite mean and the estimated grades. Trend plots were then prepared for each estimation domain-variable-combinations to ensure sample trends had been maintained. No discrepancies were observed.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Dry bulk density was applied to the Mineral Resource assigned based on the interpreted weathering. The reported Mineral Resource tonnage is considered dry.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• The adopted cut-off grade of 1.5% TREO was the minimum grade expected to support mining at Monte Muambe.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• The mineralisation is assumed to be amenable to open pit mining exclusively. • Reasonable prospects for eventual economic extraction have been constrained by open pit optimization which used an estimate of the TREO price of \$24.65 per kg of TREO and 3% of royalties applied. The processing costs were split into beneficiation (\$25/ROM t) and hydrometallurgical plant (\$66/t of product) and the overall recovery (beneficiation and hydrometallurgical plant) percentage was 48%. An overall angle of 43 degrees was used for target 1 and 6 area, and 47 degrees for target 4, based in a study of a similar mine. • The rock types were divided into free dig material and drill and blast which have different mining costs. • The annual ore output expected was of 750 000 t and s discount rate of 10%. • The mining dilution considered is 5% and mining recovery of 95%.

Criteria	JORC Code (2012) explanation	Commentary
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• Two 30 kg samples representative of low-grade mineralization and of high-grade mineralization were prepared and sent to Auralia Metallurgy in Perth for metallurgical testing. Preliminary beneficiation tests included size by assay analysis, comminution, gravity separation, flotation, magnetic separation and hydrometallurgy. • The purpose of the beneficiation testwork program was a simple sighter program to give an indication as to likely beneficiation routes that can be used and where future focus may lie on metallurgical flowsheet development. This metallurgical testwork is ongoing. • Both beneficiation and hydrometallurgical flowsheet and recovery assumptions have been made based on mineralogical similarities to other deposits which are at later stages of process development or are in production.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	Mineral exploration and mining activities in Mozambique must comply with the provisions of the Environmental Law (Law no 20/1997 of 1 October), the Mining Law (Law no 20/2014 of 18 August), and the Environmental Regulations for Mining Activities (Decree no 26/2004 of 20 August).. Exploration activities on LPP7573L are carried out under an Environmental Management Plan prepared by local environmental consultancy GeoAmbiente Lda. The Company's activities were subjected to an independent Environmental Audit which was validated the National Agency for Environmental Quality Control (AQUA) of Tete Province on 24 October 2022. As part of its Mining Concession application, the Company will prepare an EMP covering the proposed mining operations, and subsequently a Level A EIA. A review of the Project's main Environmental and Social risks was done by local environmental consultancy GeoAmbiente Lda (Jamal et al, 2023)

<table>
<tr><th>Criteria</th><th>JORC Code (2012) explanation</th><th>Commentary</th></tr>
<tr><td>Bulk density</td><td>Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.
The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.
Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>Density was measured on 371 dry core samples. After reviewing available methods against the characteristics of the Monte Muambe cores, the following methods were used:
For competent cores, the Caliper Method was used on core sections cut at right angle to the core axis. The diameter of the core was measured using a caliper, and its lengths using a measuring tape. Samples were weighted on a scale having a 0.1 g accuracy.
For weathered cores with cracks, largely made of clay and/or limonite, which presented some swelling, the volume calculation was done using the nominal inner diameter of the coring bit. Samples were weighted on a scale having a 0.1 g accuracy.
The density of samples that were crumbling and had no integrity or competency was not measured.
During the Competent Person's representative's visit in August 2023, the density of 20 samples was rechecked, using a modified Caliper Method (to suit the fact that the cores were now half cores), and using the Saturated Immersion Method. The accuracy of the scale was also checked using reference weights. These checks confirmed the reliability of the original density database.
Conditional mean densities were assigned based on the interpreted weathering.
<table>
<tr><th rowspan="2">Target</th><th rowspan="2">Description</th><th rowspan="2">Interpreted Weathering</th><th>Assigned density</th></tr>
<tr><th>Density t/m3</th></tr>
<tr><td rowspan="5">1</td><td>Soil</td><td>SO</td><td>1.80</td></tr>
<tr><td>Very Weathered</td><td>VW</td><td>2.10</td></tr>
<tr><td>Weathered</td><td>W</td><td>2.55</td></tr>
<tr><td>Slightly weathered</td><td>SW</td><td>2.60</td></tr>
<tr><td>Fresh</td><td>FR</td><td>2.70</td></tr>
<tr><td rowspan="5">4</td><td>Soil</td><td>SO</td><td>1.80</td></tr>
<tr><td>Very Weathered</td><td>VW</td><td>2.10</td></tr>
<tr><td>Weathered</td><td>W</td><td>2.55</td></tr>
<tr><td>Slightly weathered</td><td>SW</td><td>2.60</td></tr>
<tr><td>Fresh</td><td>FR</td><td>2.70</td></tr>
</table></td></tr>
</table>

Criteria	JORC Code (2012) explanation	Commentary

Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.*
	• *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).*
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.*
		• Classification of the Mineral Resource incorporated confidence in the:
		  • Available geological and sample data
		  • The geological knowledge and interpretation
		  • The confidence in the demonstrated geological and grade continuity
		  • Confidence in the final Mineral Resource estimate; and
		  • Results of the open pit optimization.
		• All relevant factors have been taken into account.
		• The resource categories accurately reflect the confidence of the Competent Person in the deposit.

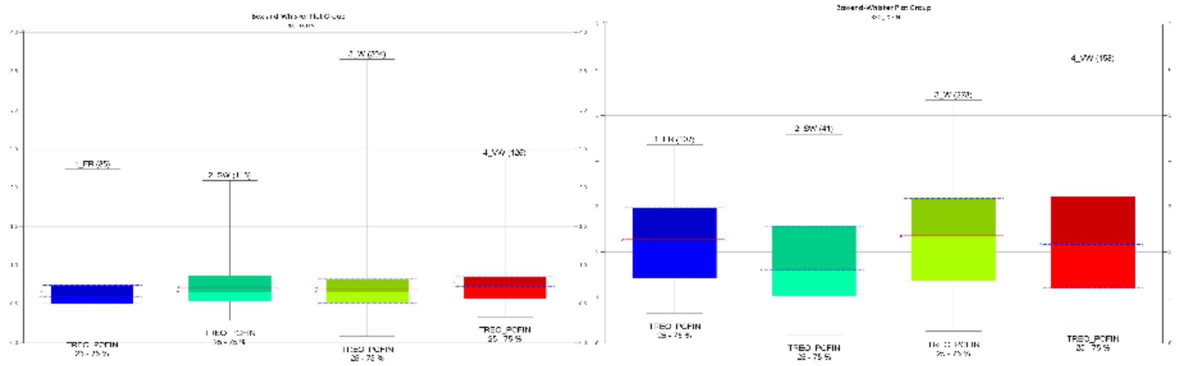
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• The 2023 Mineral Resource estimate has been reviewed internally. No external review has been undertaken.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The 2023 Mineral Resource estimate is a global estimate with confidence being commensurate with the Mineral Resource classification (primarily Inferred and Indicated Mineral Resource) and drill hole spacing. • The 2023 Monte Muambe Mineral Resource is a maiden resource and it has no production history.

Appendix C

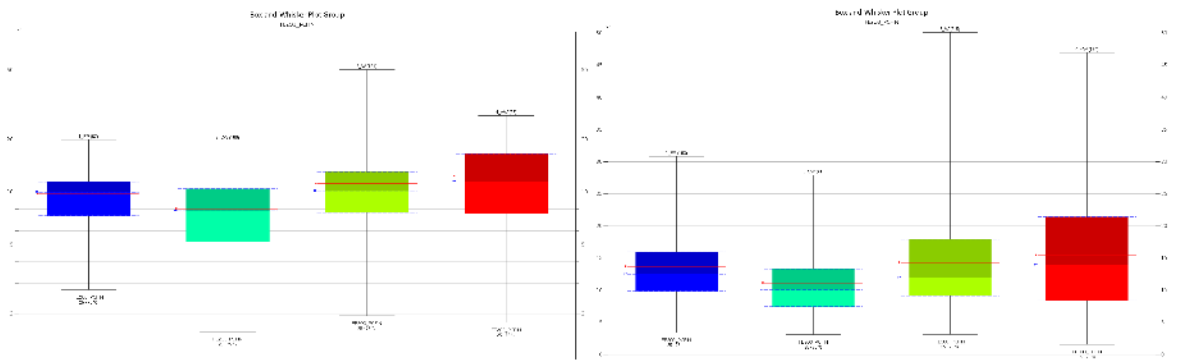
Box and whisker plots; key grade variables

Target 1 Box and whisker plots by weathering (left ESTDOM T1_01, right ESTDOM T1_11)

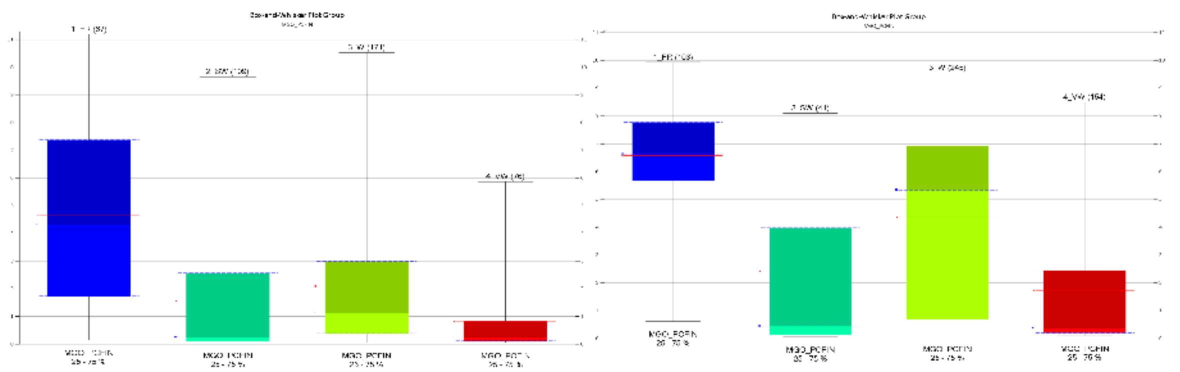
TREO %



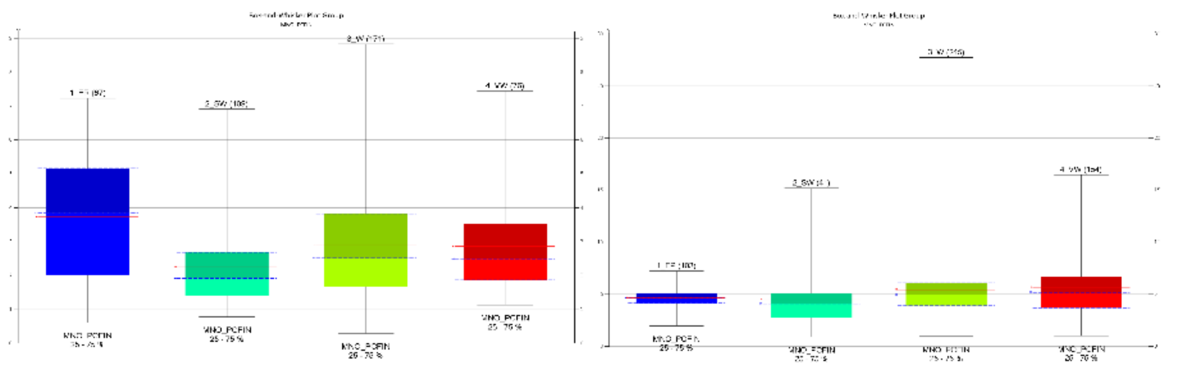
Fe_2O_3 %



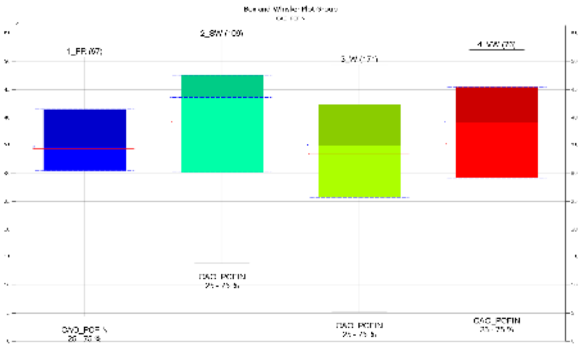
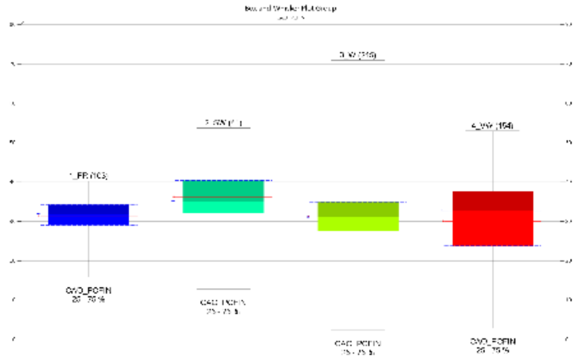
MgO %



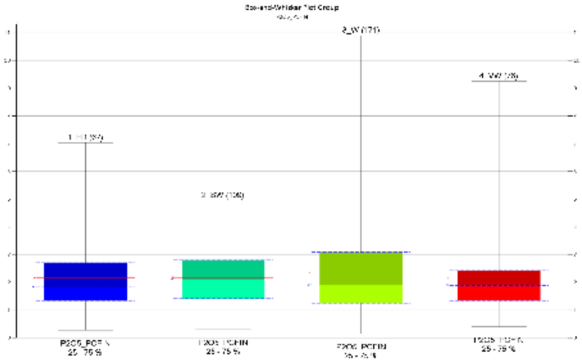
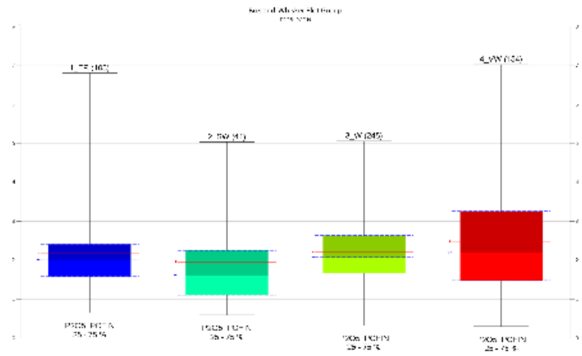
MnO %



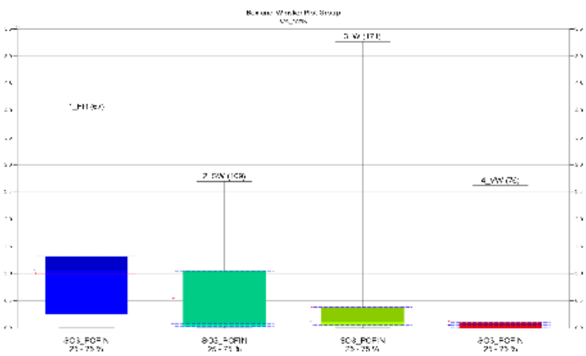
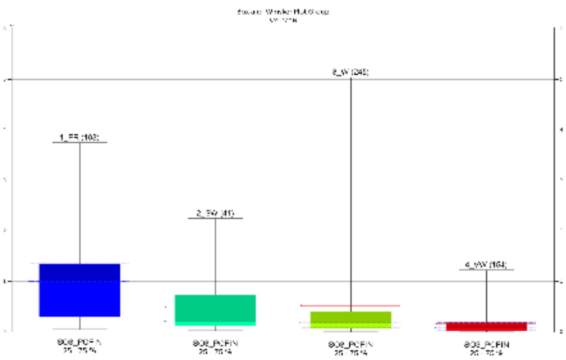
CaO %

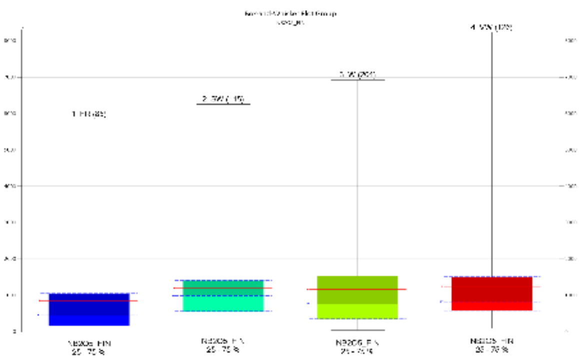
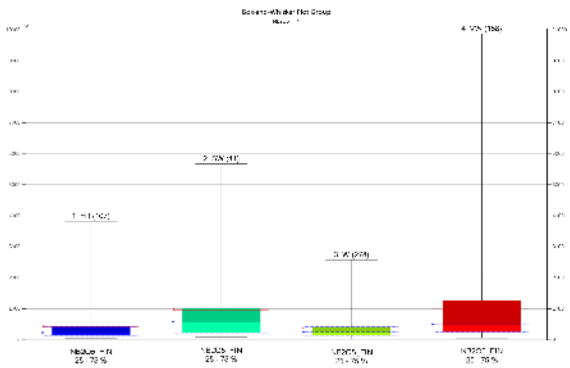
P_2O_5 %

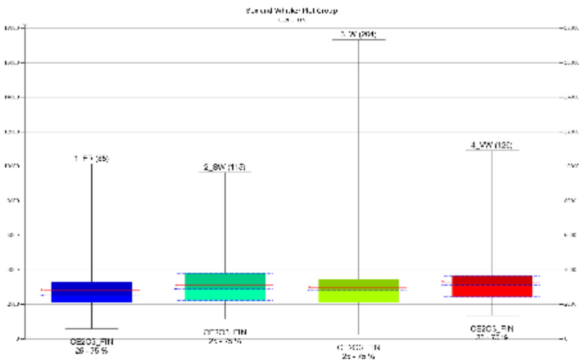
SO_3 %

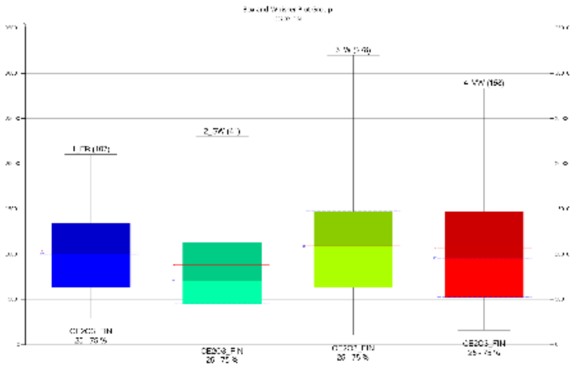



Nb_2O_5 ppm

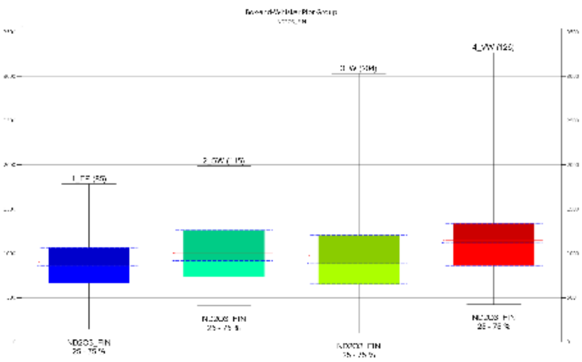



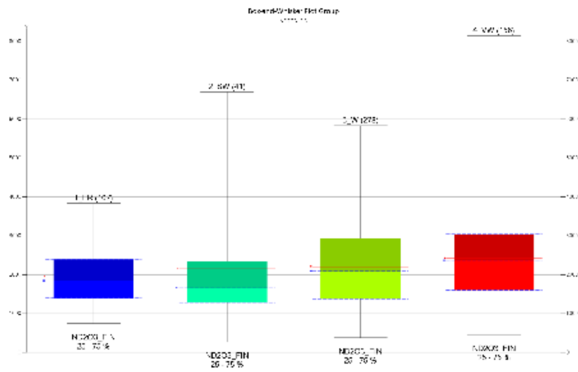
Ce_2O_3 ppm



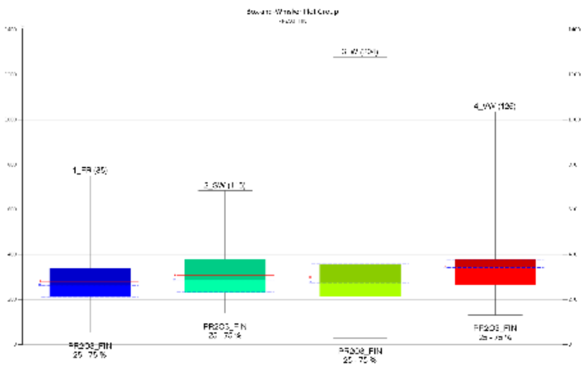


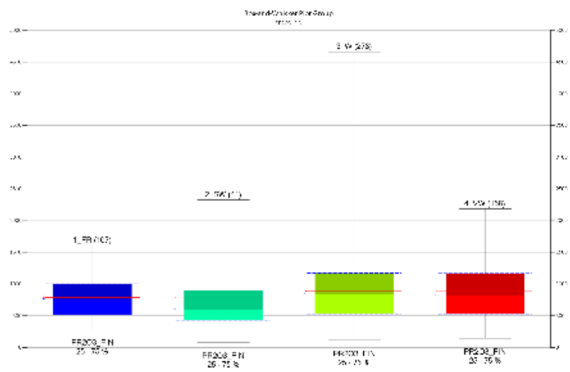
Nd_2O_3 ppm



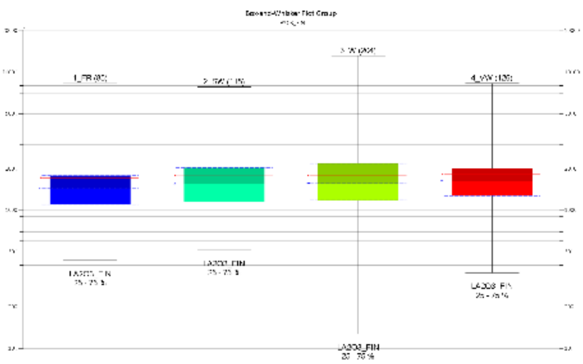


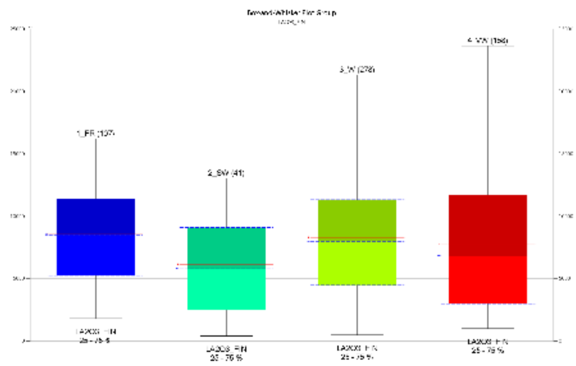
Pr_2O_3 ppm



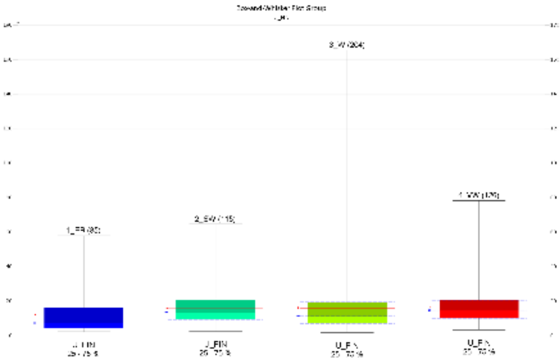


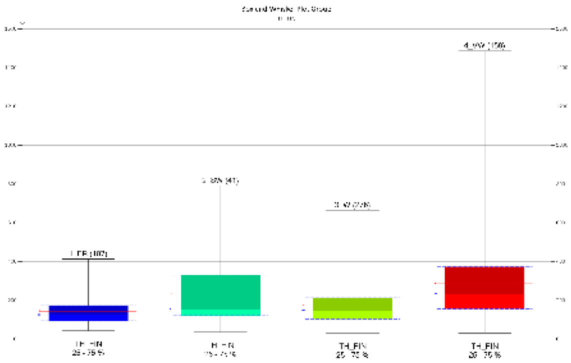
La_2O_3 ppm



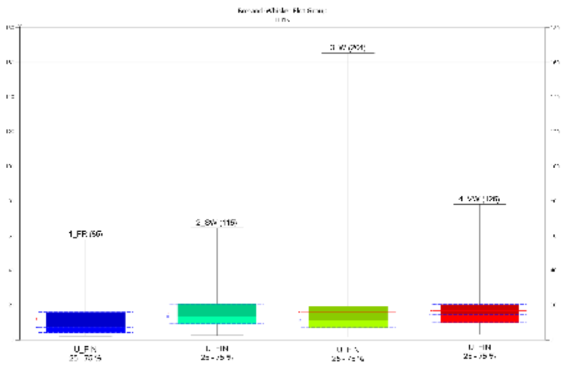


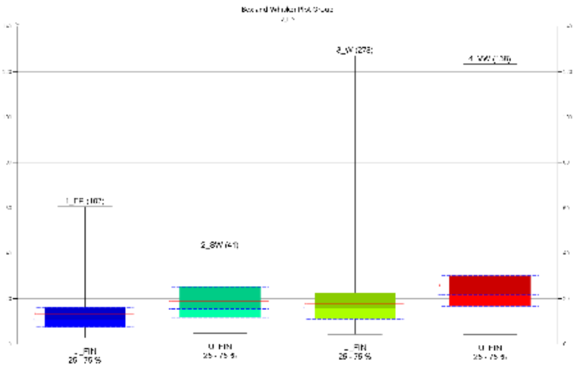
Th ppm





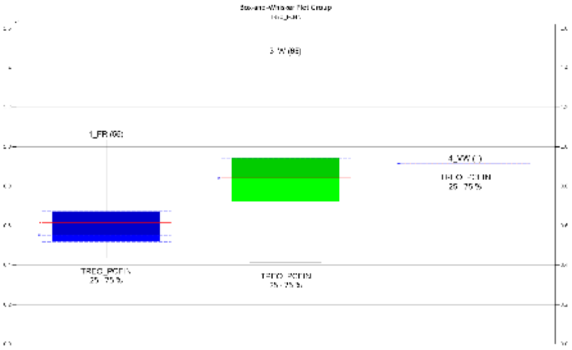
U ppm

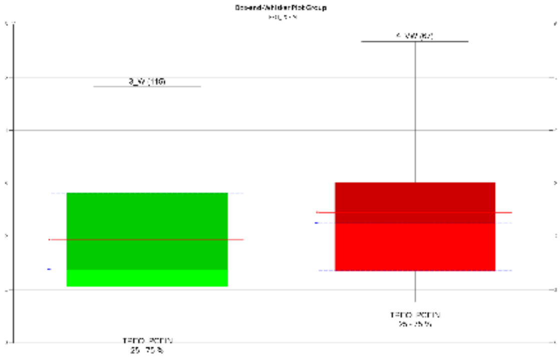




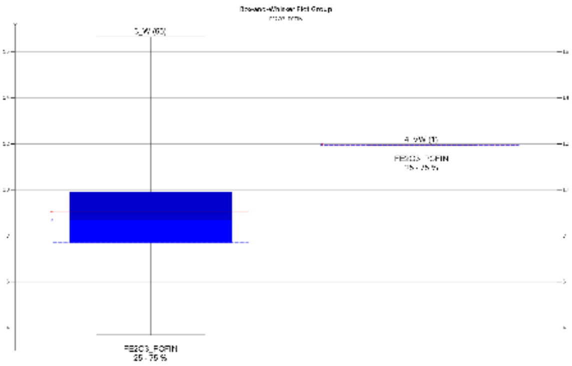
Target 4 Box and whisker plots by weathering (left ESTDOM T1_01, right ESTDOM T1_11)

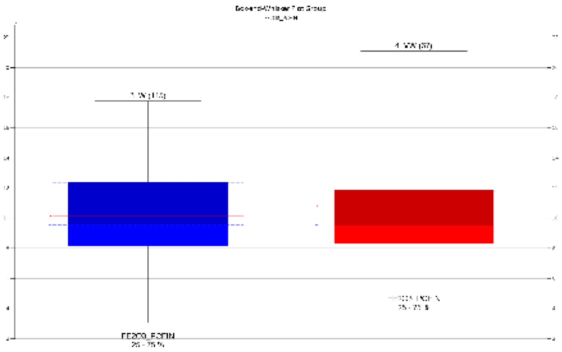
TREO %



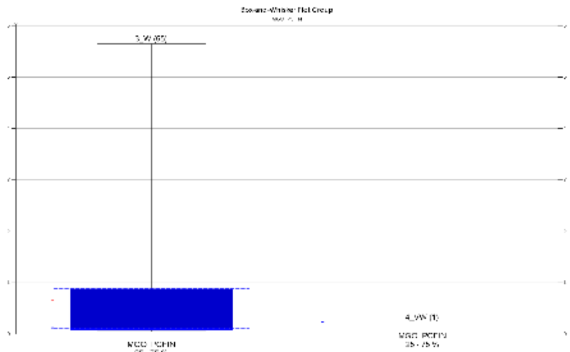


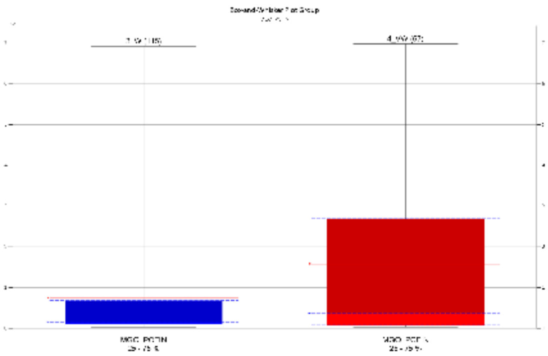
Fe_2O_3 %



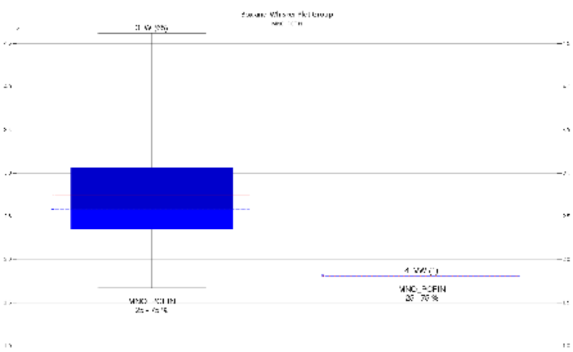


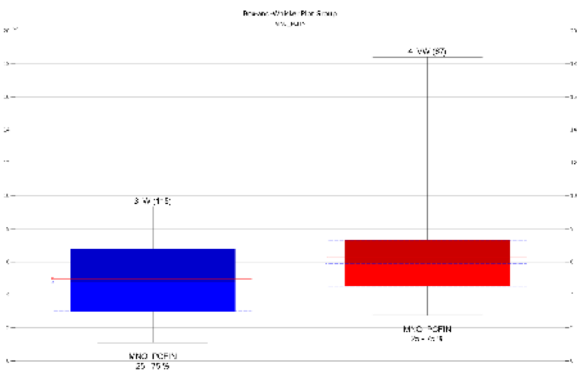
MgO %



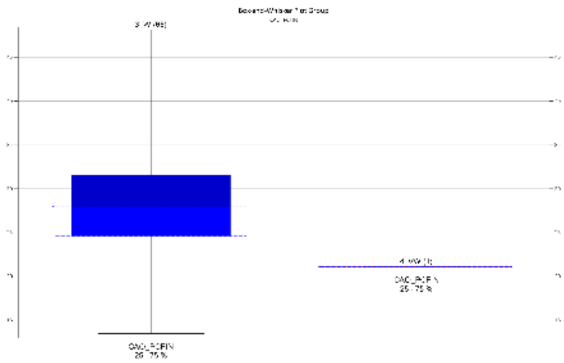


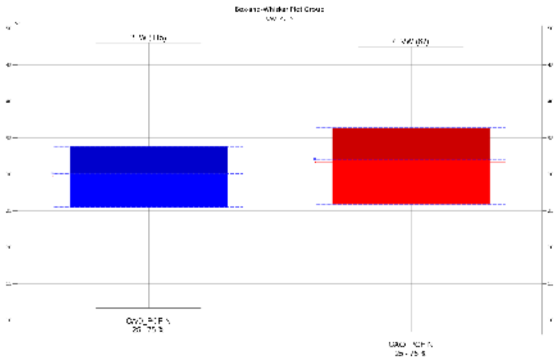
MnO %



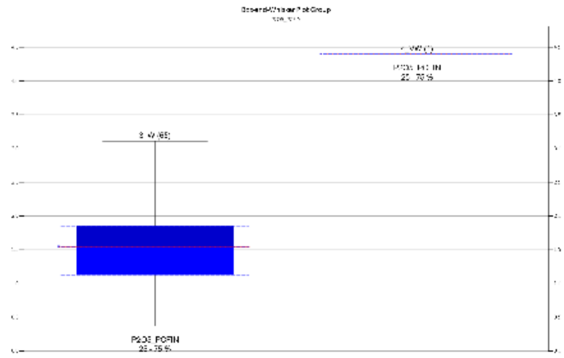


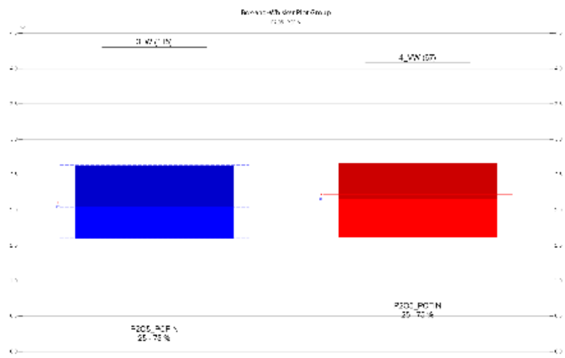
CaO %



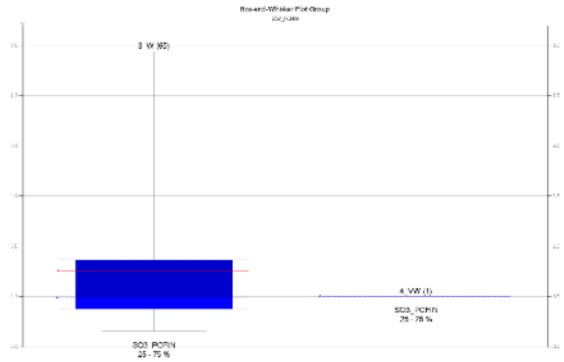


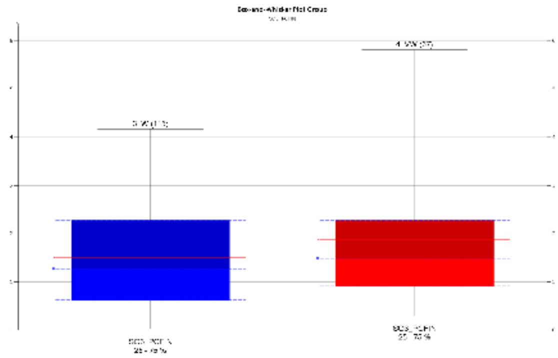
P_2O_5 %



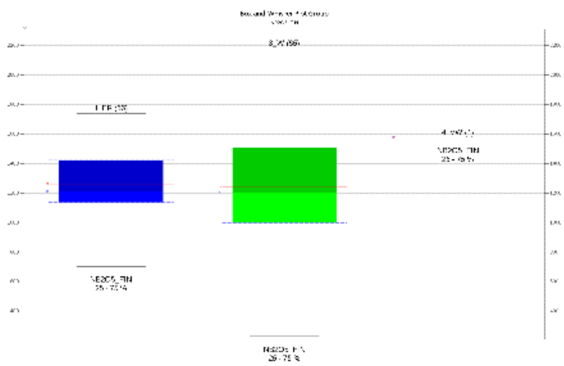


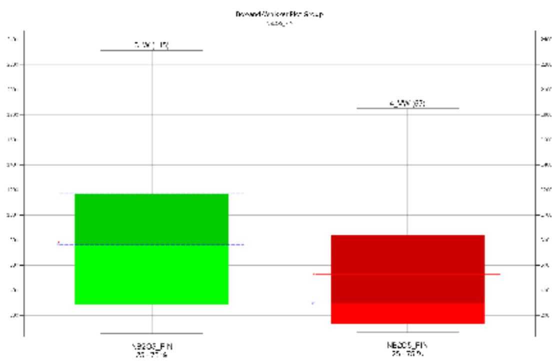
SO_3 %



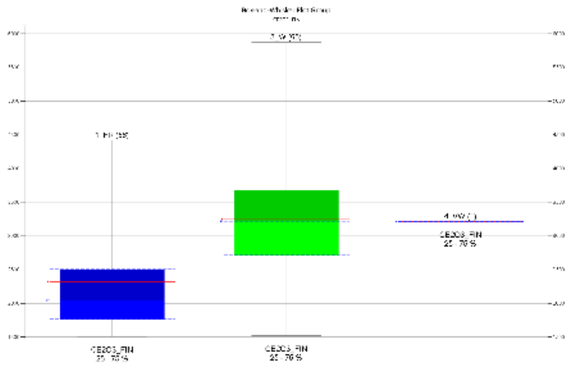


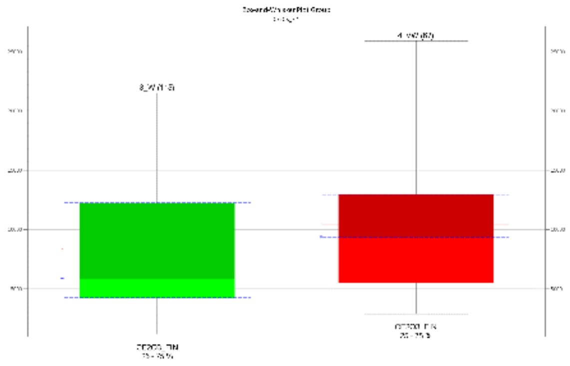
Nb_2O_5 ppm



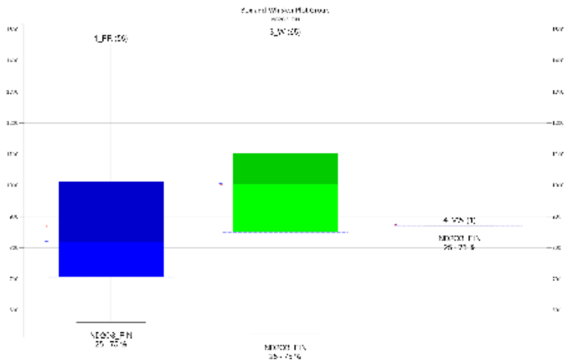


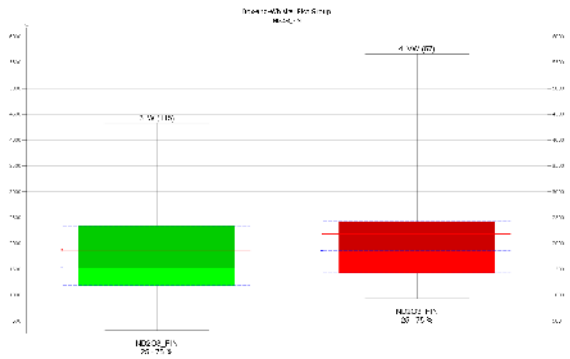
Ce_2O_3 ppm



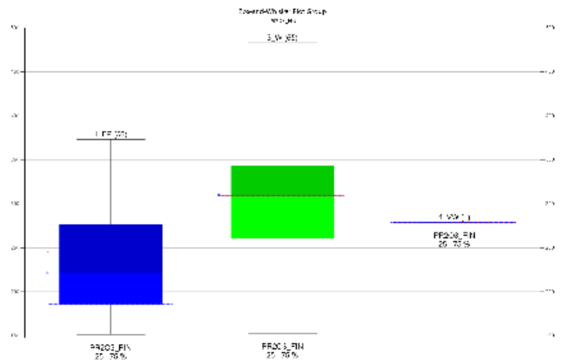


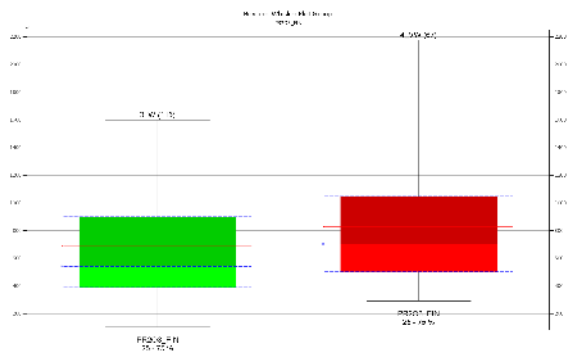
Nd_2O_3 ppm



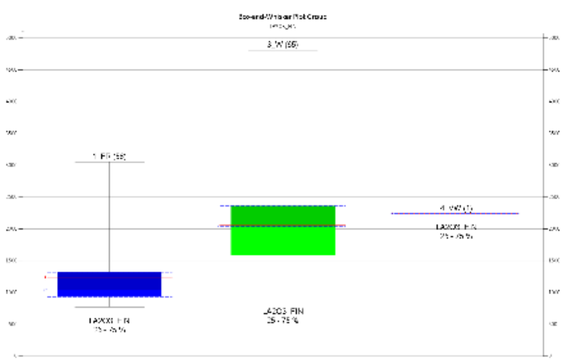


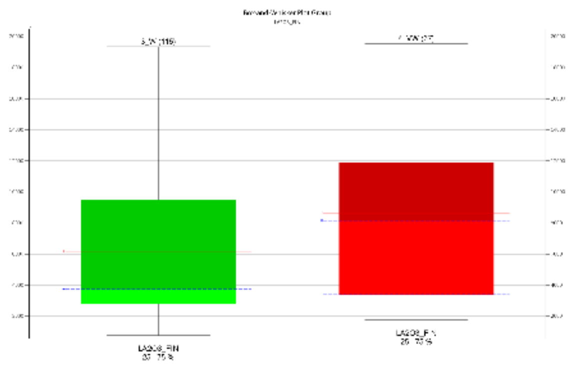
Pr_2O_3 ppm



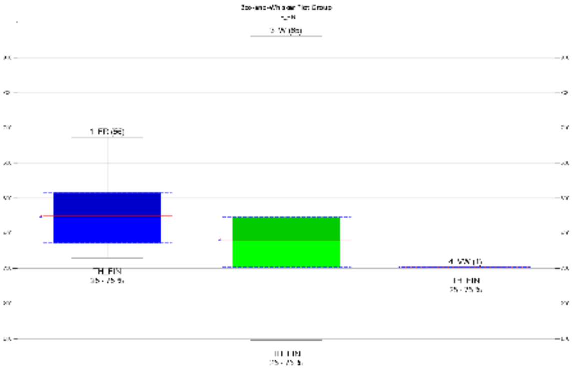


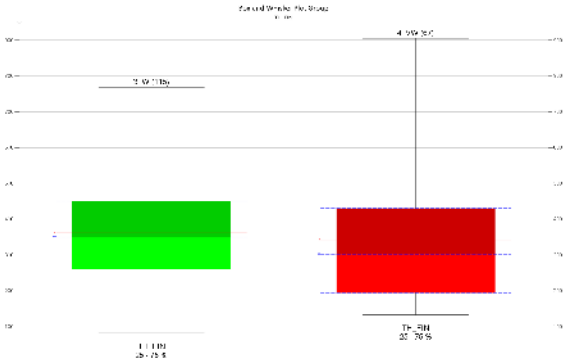
La_2O_3 pp



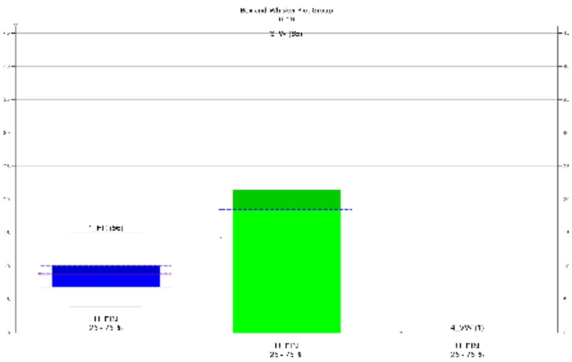


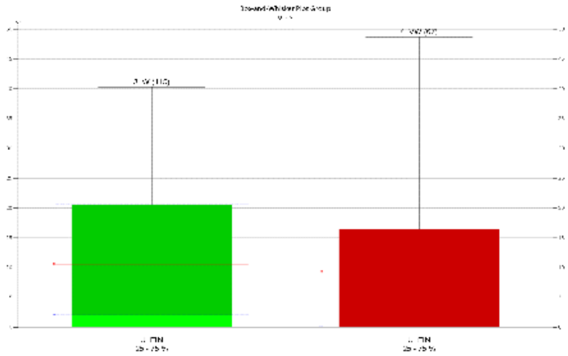
Th ppm





U ppm

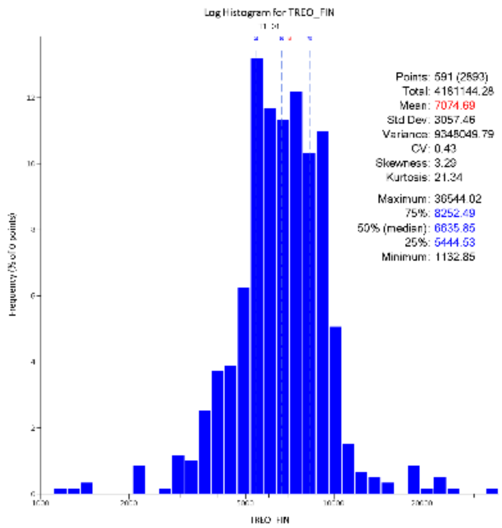


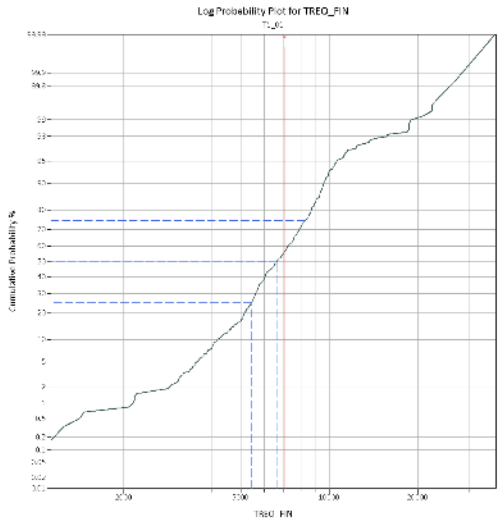


Appendix D
Grade distribution by domain

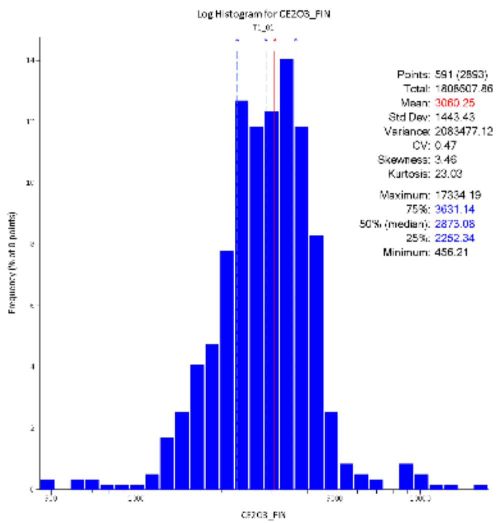
Target 1 Log-plot histogram and probability plots 0.5 to 1.0% TREO domain (ESTDOM T1_01)

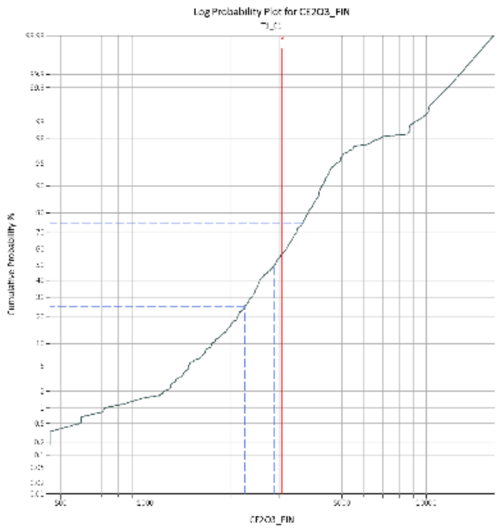
TREO



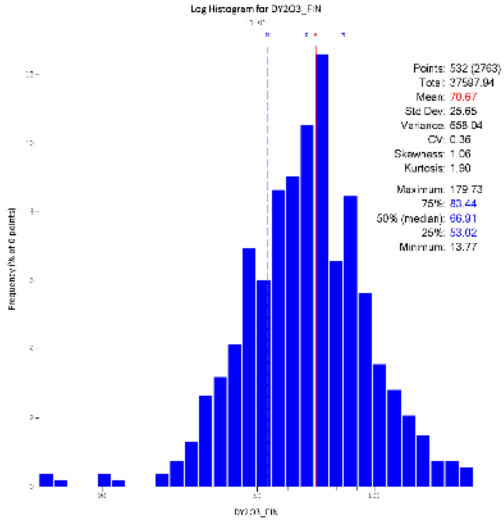


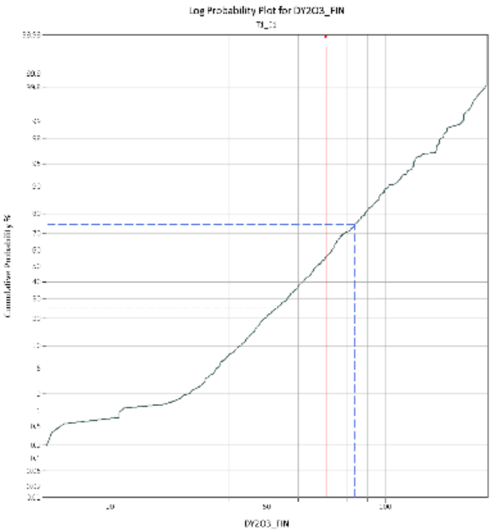
Ce_2O_3



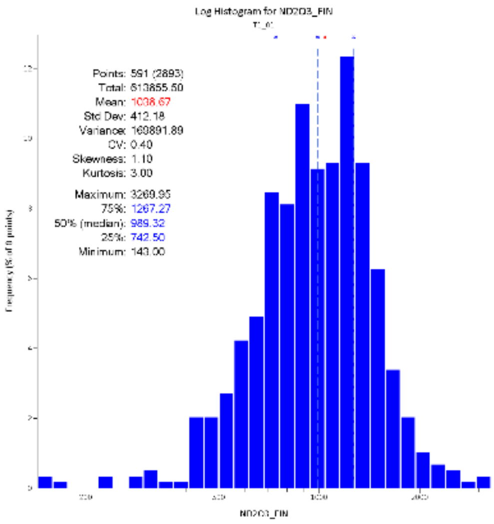


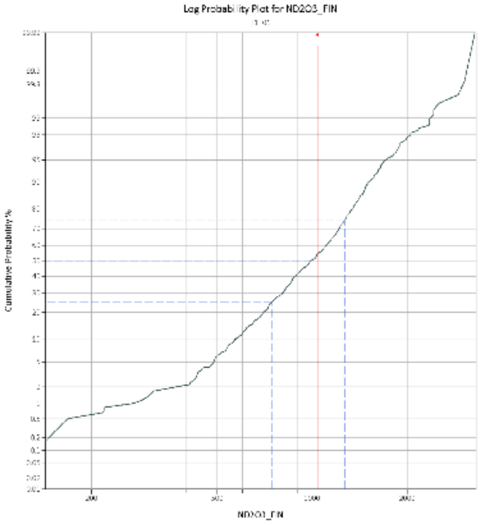
Dy_2O_3



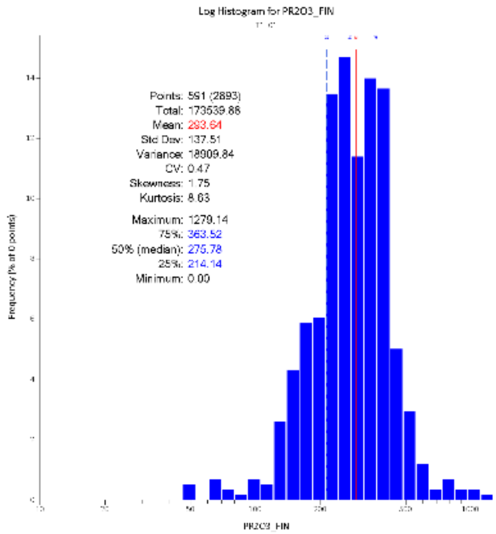


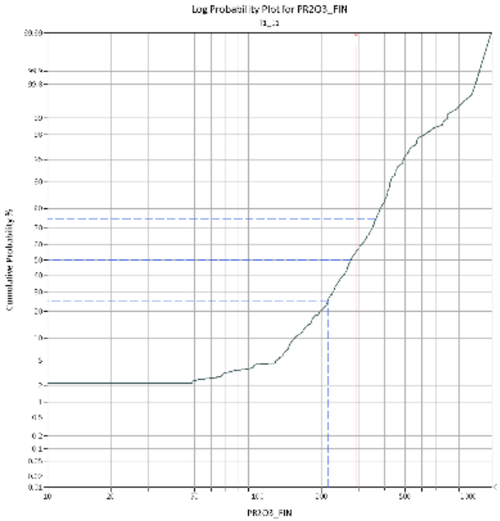
Nd_2O_3



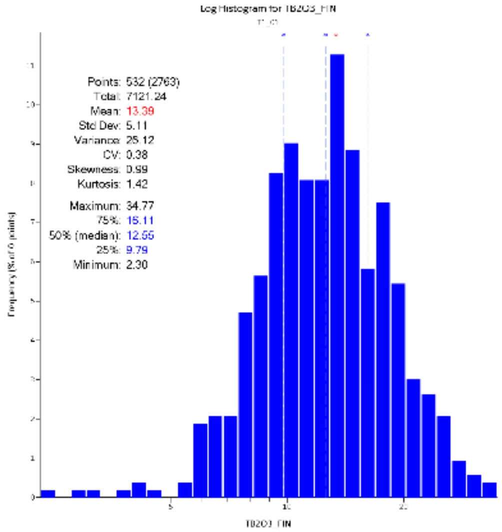


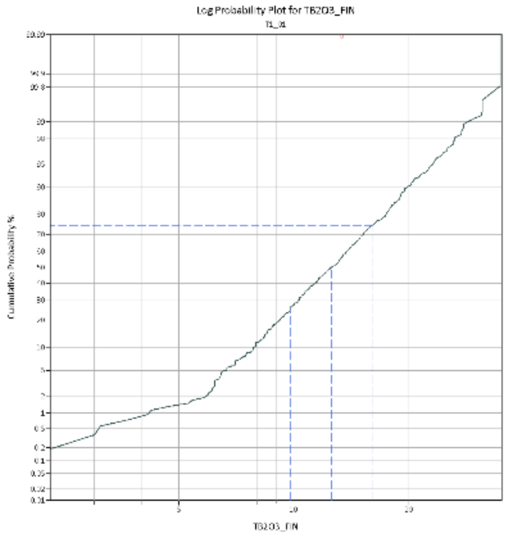
Pr_2O_3



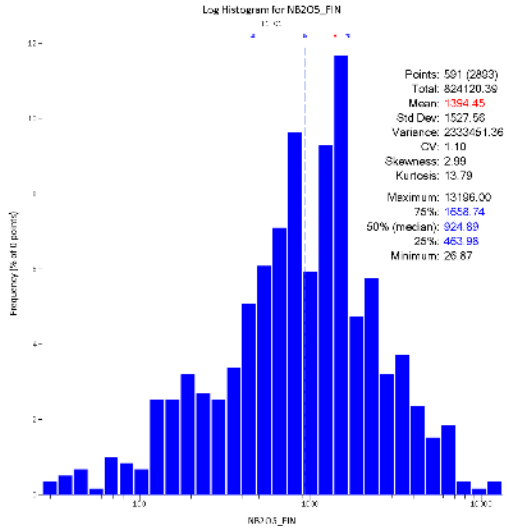


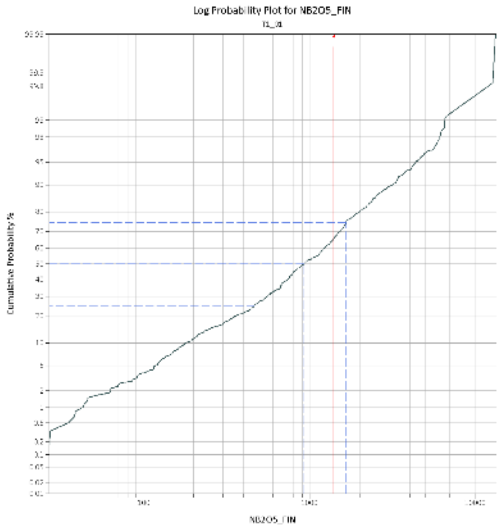
Tb_2O_3



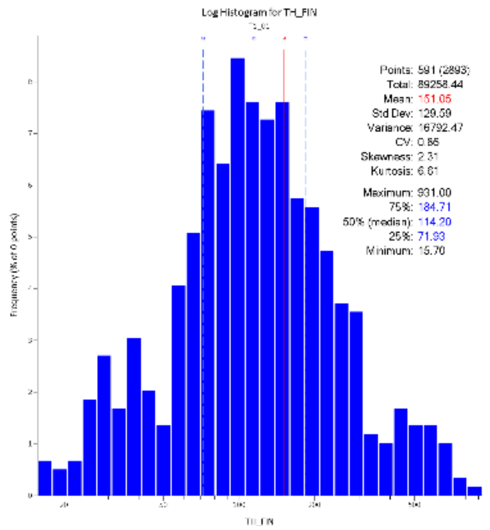


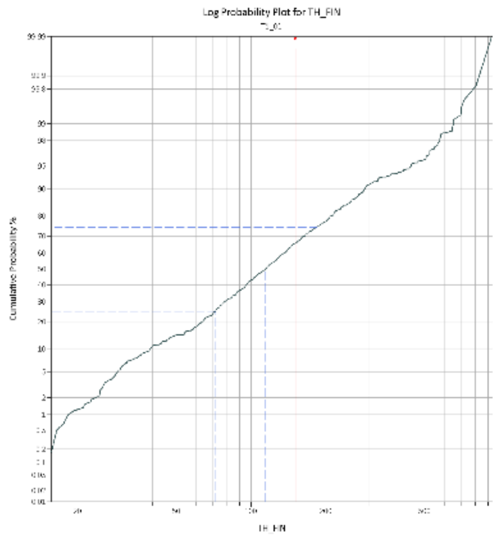
Nb_2O_5



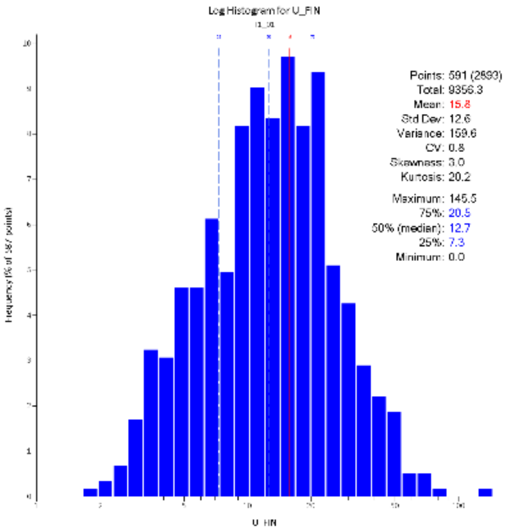


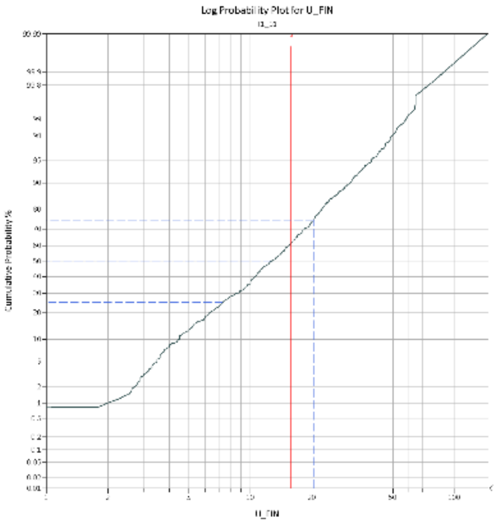
Th





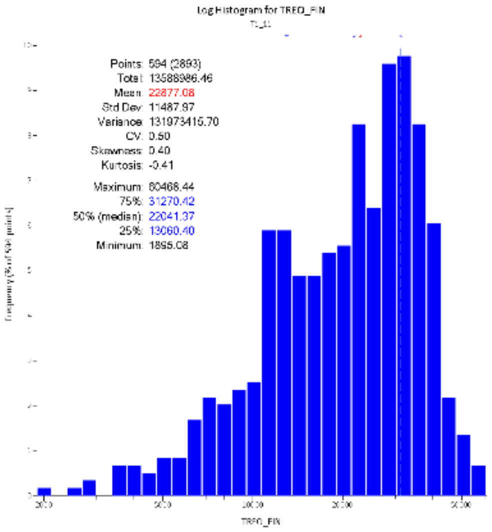
U

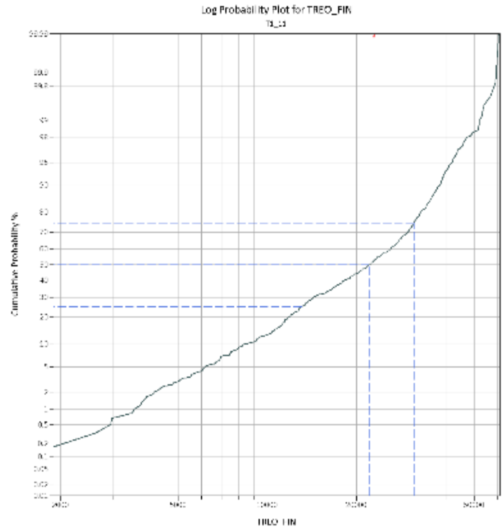

Log Histogram for U_FIN
Points: 591 (2893)
Total: 9356.3
Mean: 15.8
Std Dev: 12.6
Variance: 159.6
CV: 0.8
Skewness: 3.0
Kurtosis: 20.2
Maximum: 145.5
75%: 20.5
50% (median): 12.7
25%: 7.3
Minimum: 0.0


Log Probability Plot for U_FIN
U_FIN

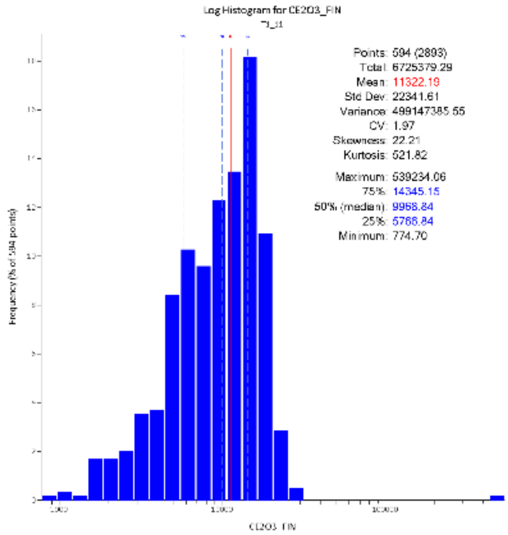
Target 1 Log-plot histogram and probability plots 0.5 to 1.0% TREO domain (ESTDOM T1_11)

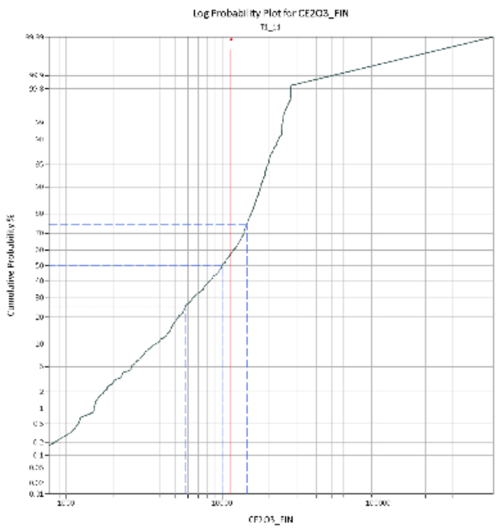
TREO



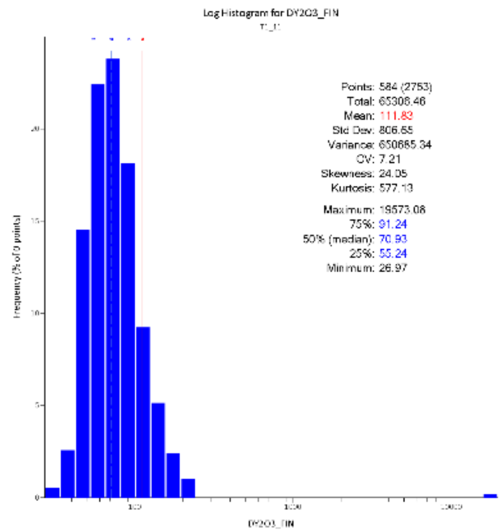


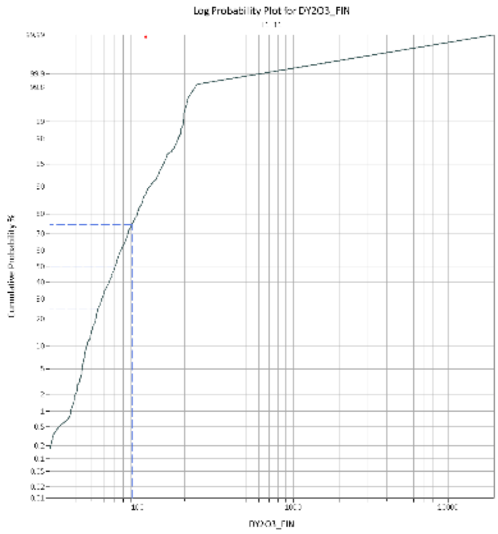
Ce_2O_3



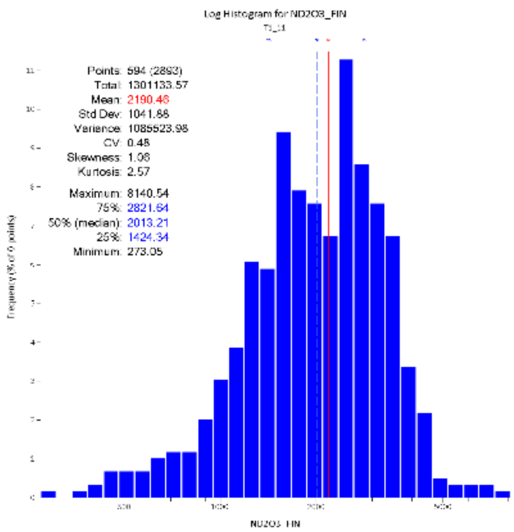


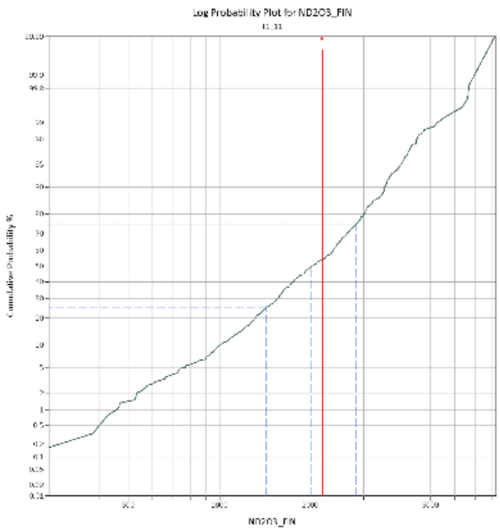
Dy_2O_3



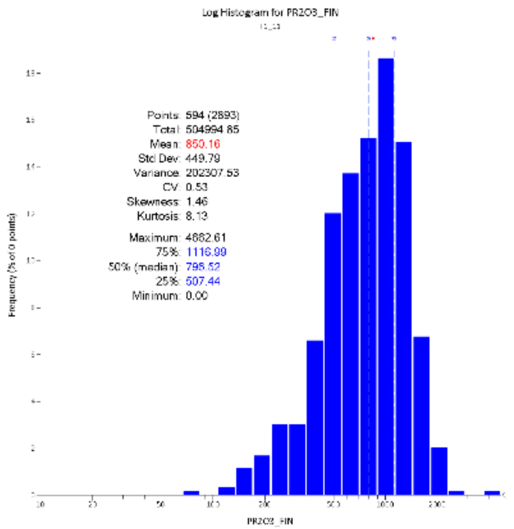


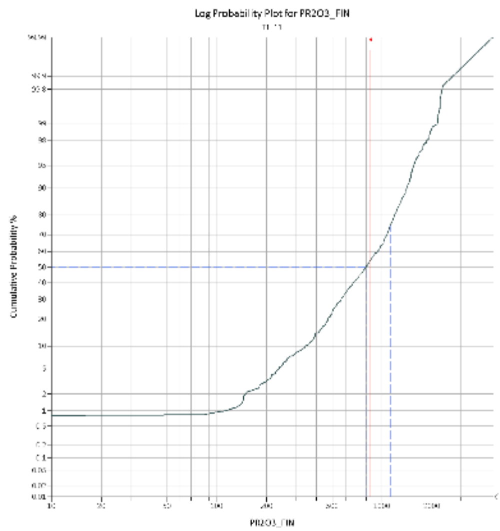
Nd_2O_3



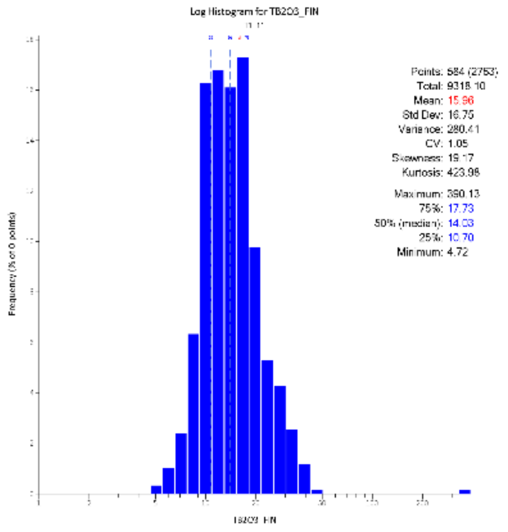


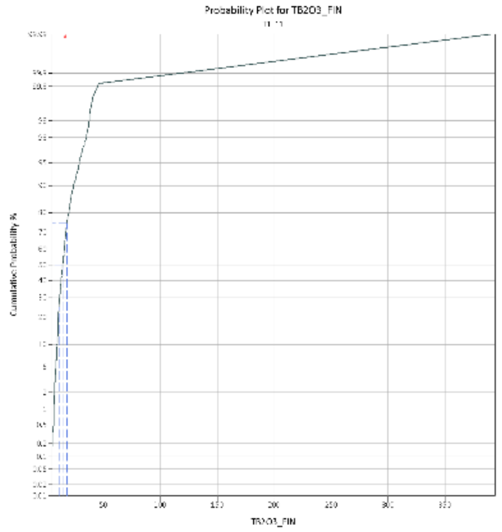
Pr_2O_3



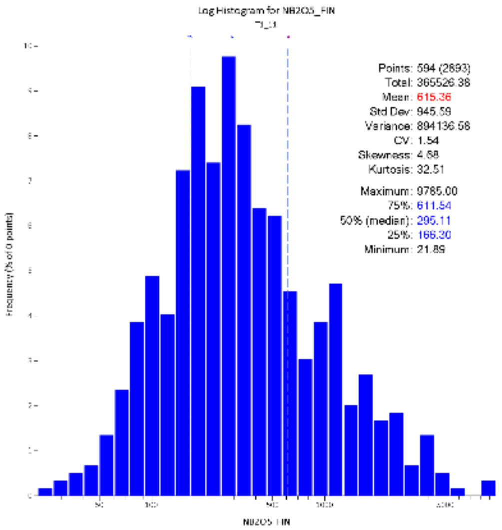


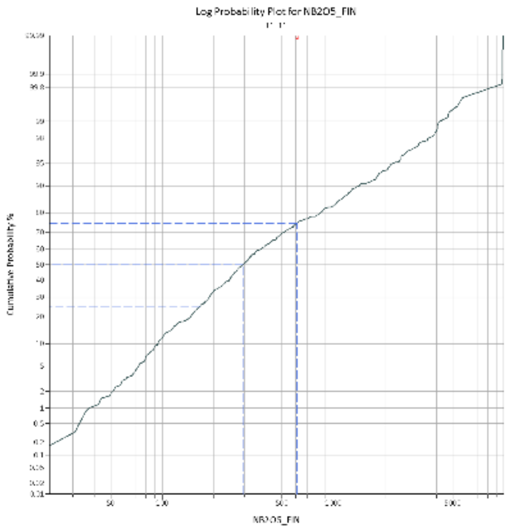
Tb_2O_3



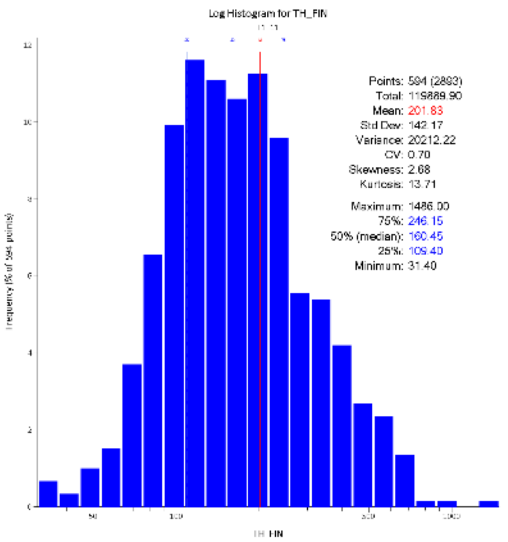


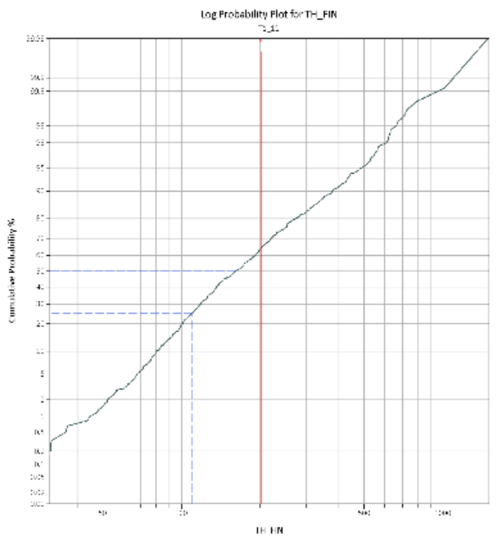
Nb_2O_5



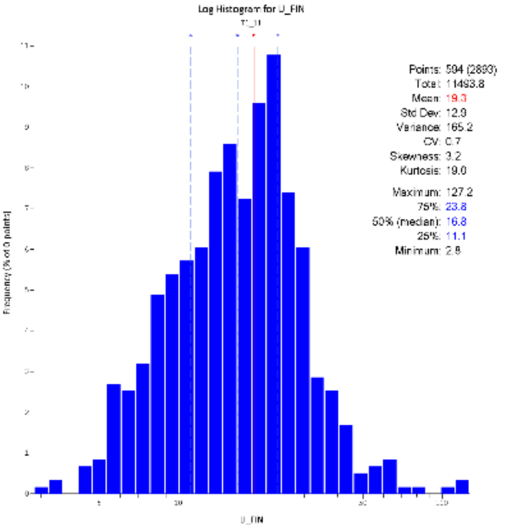


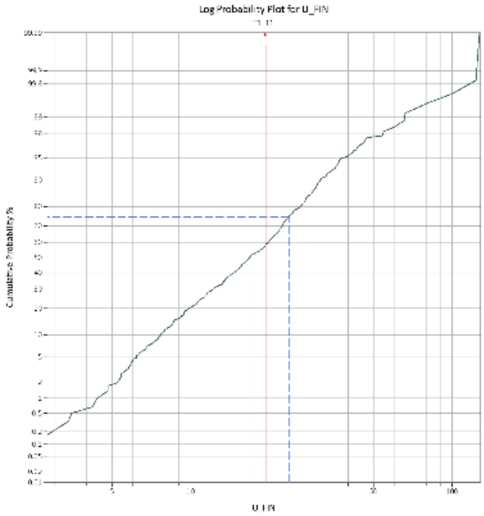
Th





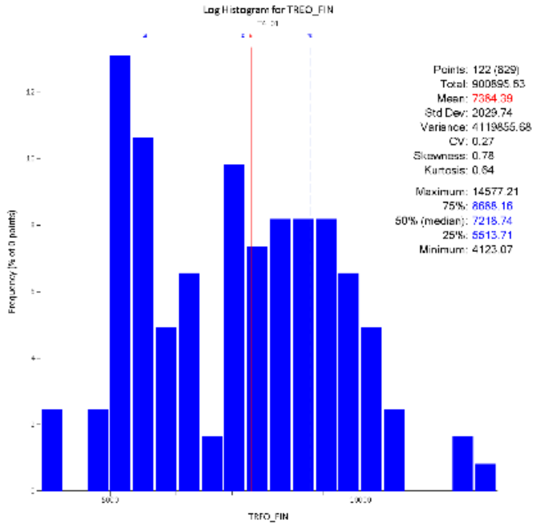
U

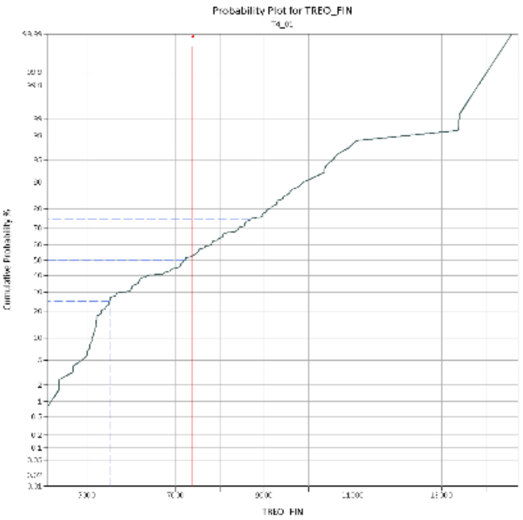

Log Histogram for U_FIN
Points: 594 (2893)
Total: 11493.8
Mean: 19.3
Std Dev: 12.9
Variance: 165.2
CV: 0.7
Skewness: 3.2
Kurtosis: 19.0
Maximum: 127.2
75%: 23.8
50% (median): 16.8
25%: 11.1
Minimum: 2.8
U_FIN


Log Probability Plot for U_FIN
U_FIN

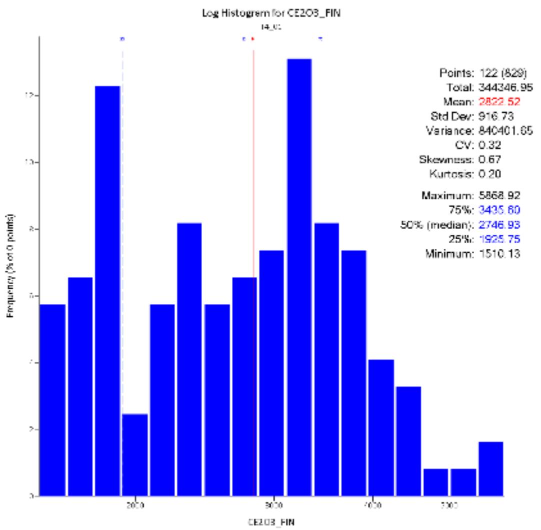
Target 4 Log-plot histogram and probability plots 0.5 to 1.0% TREO domain (ESTDOM T4_01)

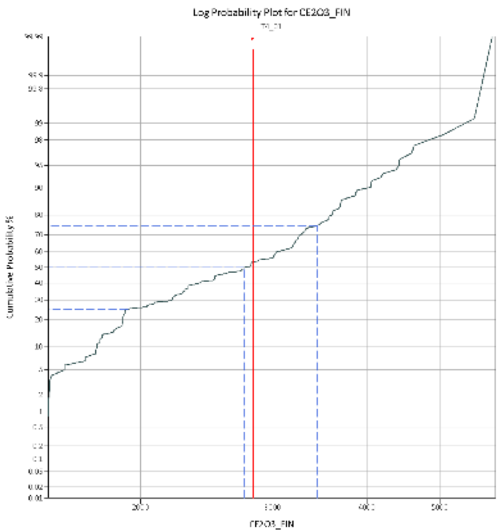
TREO



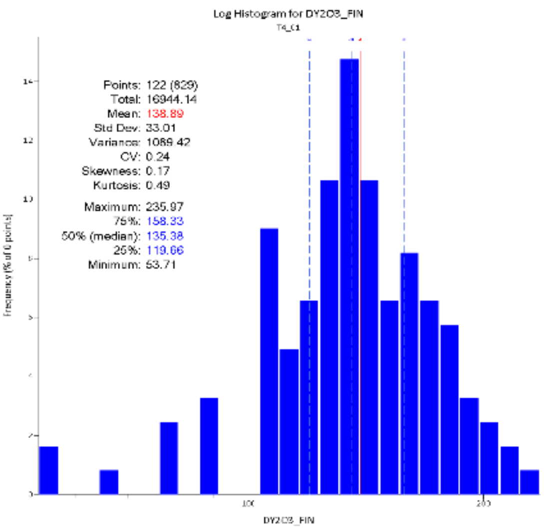


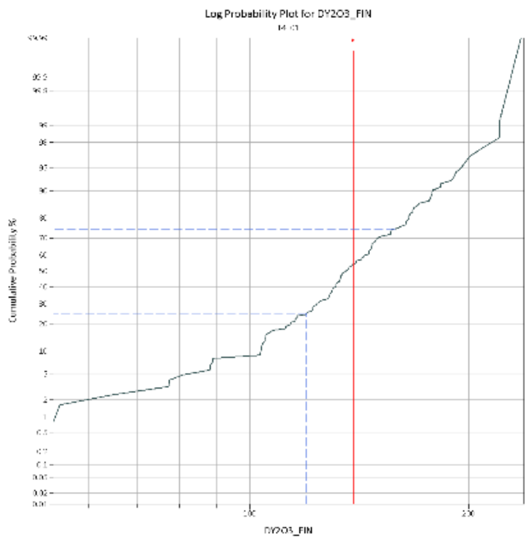
Ce2O3



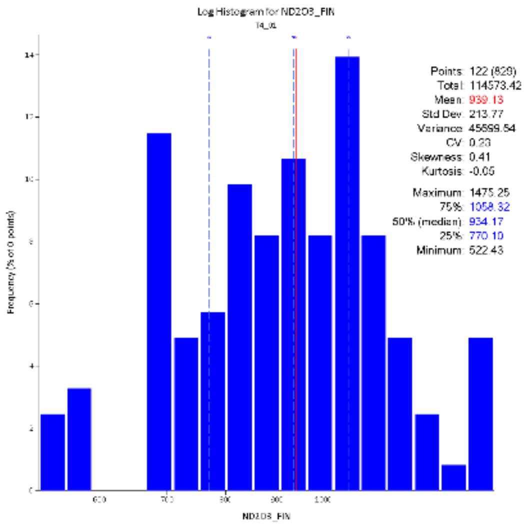


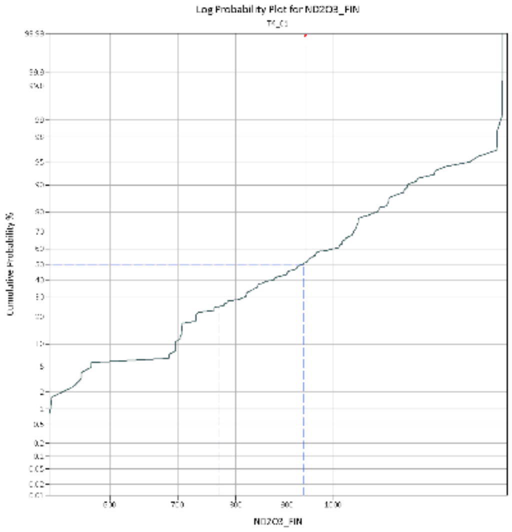
DY2O3



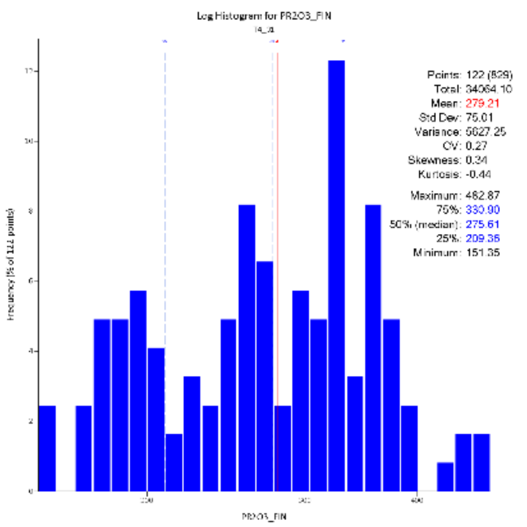


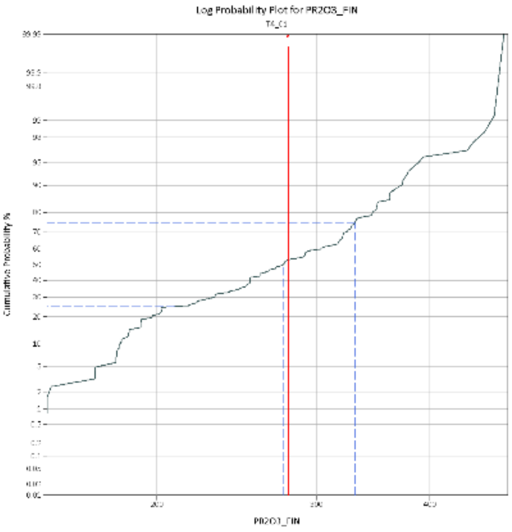
Nd2O3


Log Histogram for ND2O3_FIN
Points: 122 (829)
Total: 114573.42
Mean: 939.13
Std Dev: 213.77
Variance: 45699.54
CV: 0.23
Skewness: 0.41
Kurtosis: -0.05
Maximum: 1475.25
75%: 1058.32
50% (median): 934.17
25%: 770.10
Minimum: 522.43
ND2O3_FIN

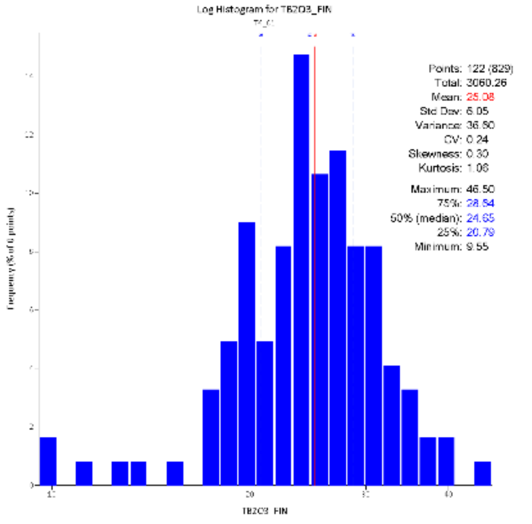

Log Probability Plot for ND2O3_FIN
ND2O3_FIN

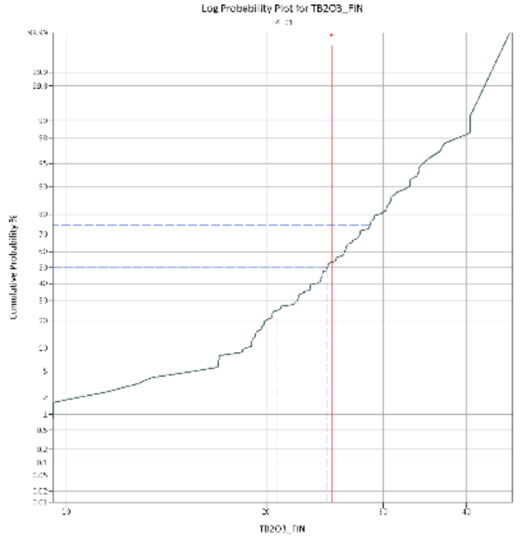
Pr2O3


Log Histogram for PR2O3_FIN
Points: 122 (829)
Total: 34064.10
Mean: 279.21
Std Dev: 75.01
Variance: 5627.25
CV: 0.27
Skewness: 0.34
Kurtosis: -0.44
Maximum: 482.87
75%: 330.90
50% (median): 275.61
25%: 209.36
Minimum: 151.35
PR2O3_FIN

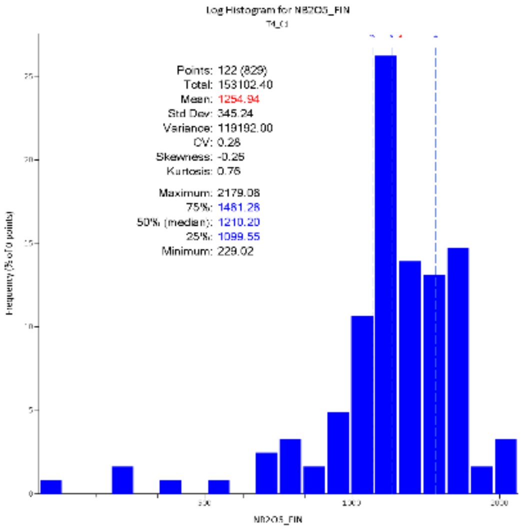

Log Probability Plot for PR2O3_FIN
PR2O3_FIN

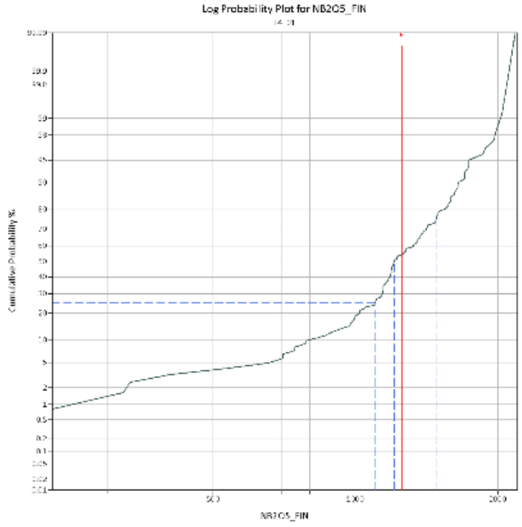
Tb2O3



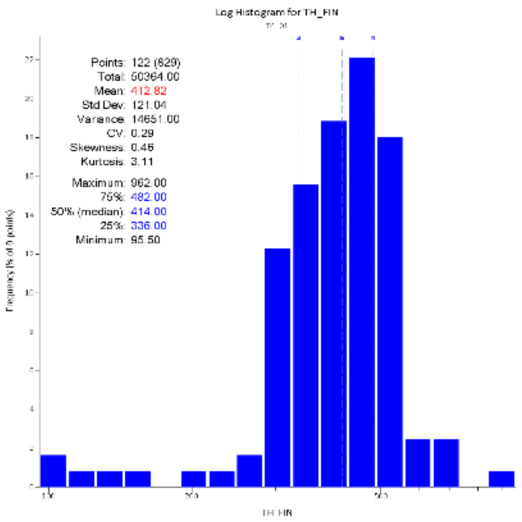


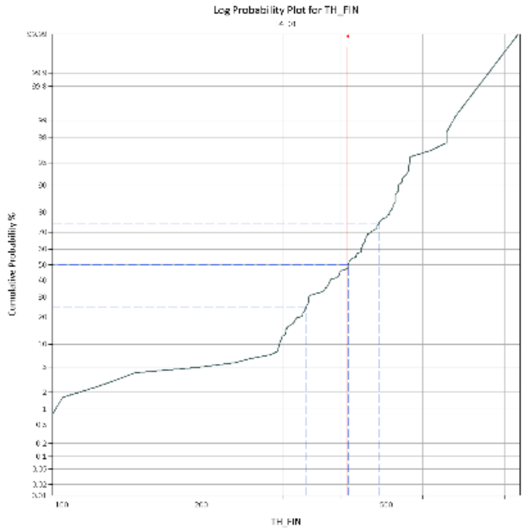
Nb2O5



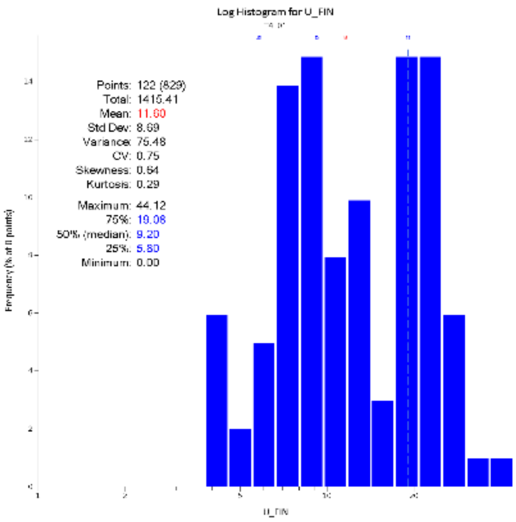


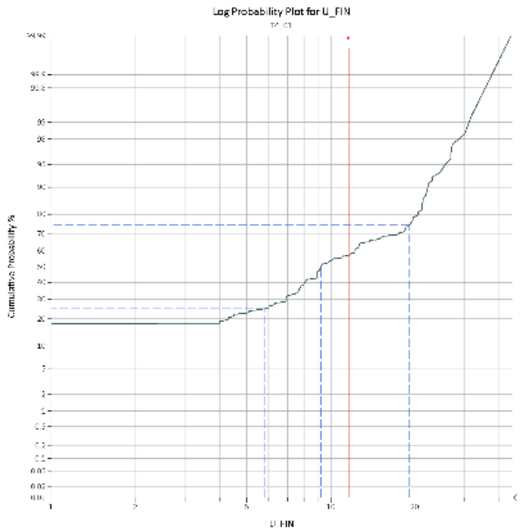
Th





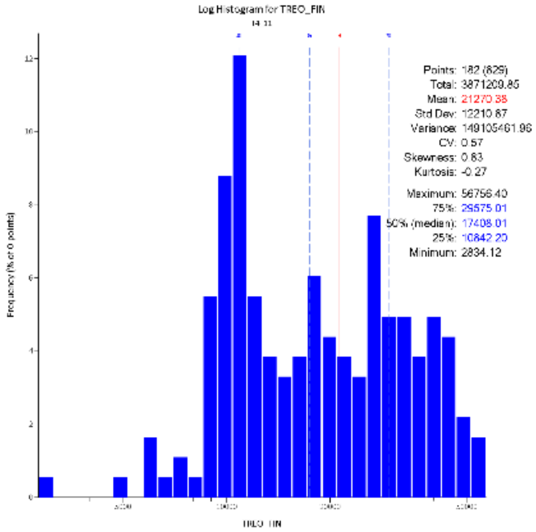
U

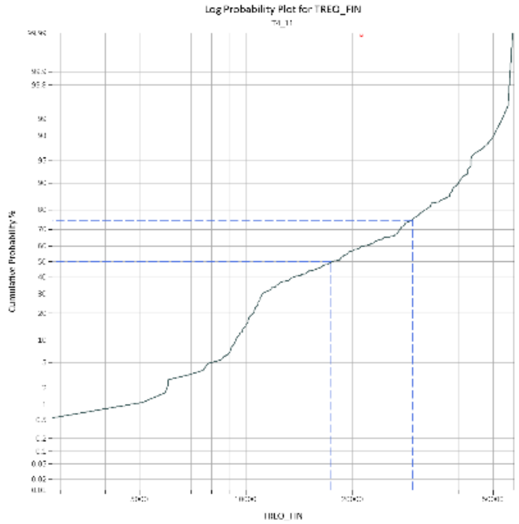

Log Histogram for U_FIN
Points: 122 (829)
Total: 1415.41
Mean: 11.60
Std Dev: 8.69
Variance: 75.48
CV: 0.75
Skewness: 0.64
Kurtosis: 0.29
Maximum: 44.12
75%: 19.08
50% (median): 9.20
25%: 5.80
Minimum: 0.00
Frequency (% of 8 points)
U_FIN


Log Probability Plot for U_FIN
Cumulative Probability %
U_FIN

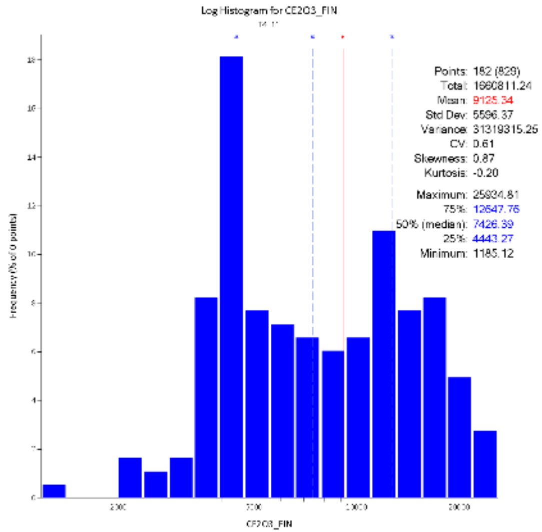
Appendix 3d – Target 4 Log-plot histogram and probability ≥1.0% TREO domain (ESTDOM T4_11)

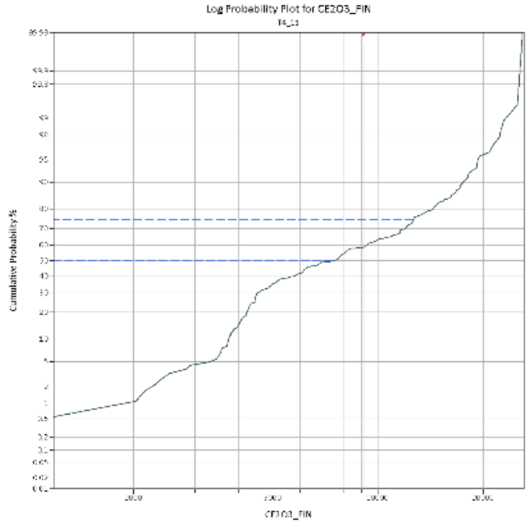
TREO



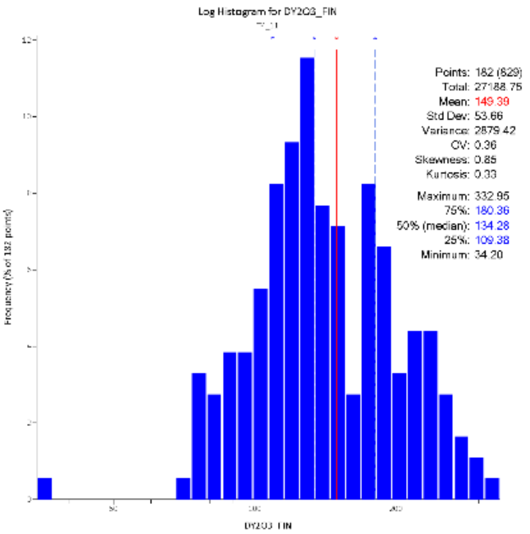


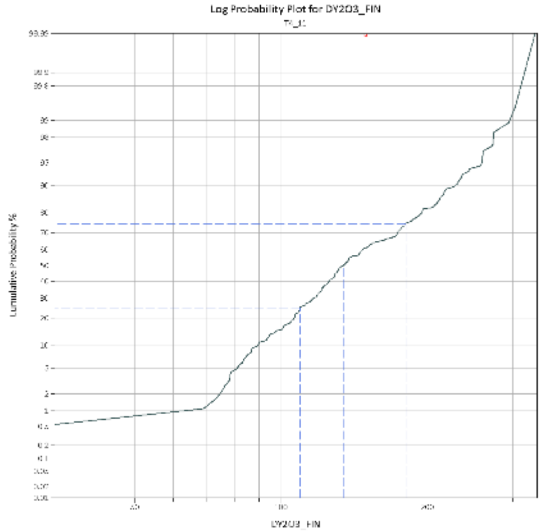
Ce2O3



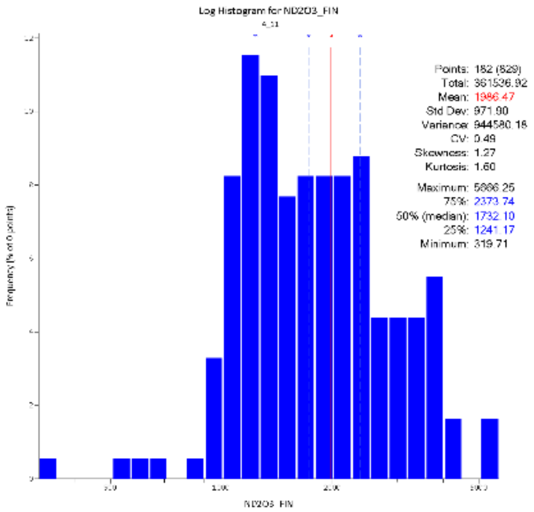


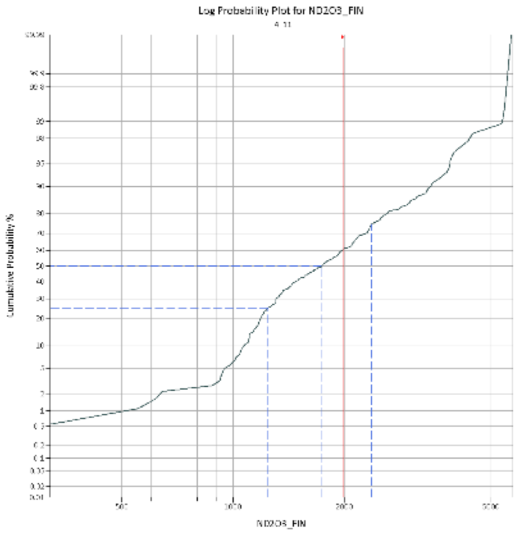
DY2O3



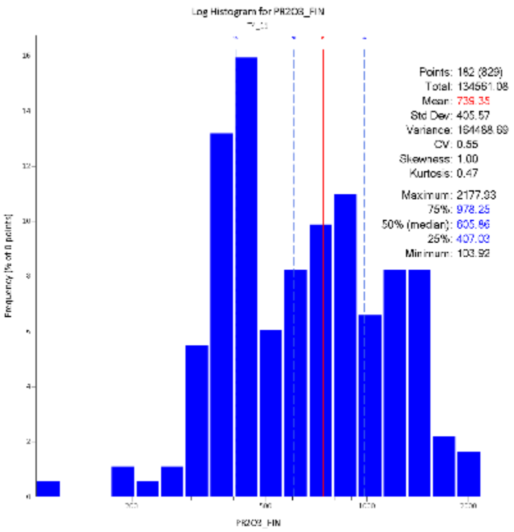


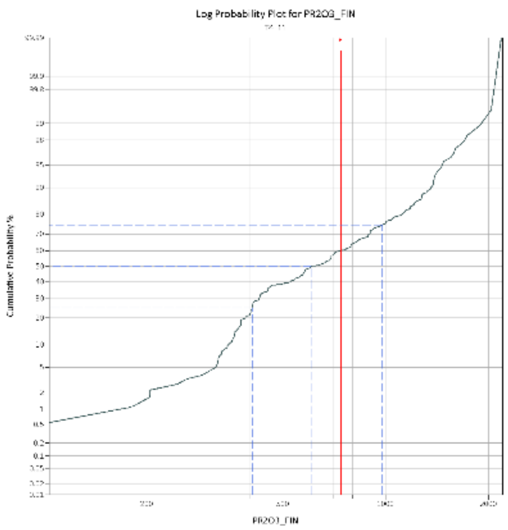
Nd2O3



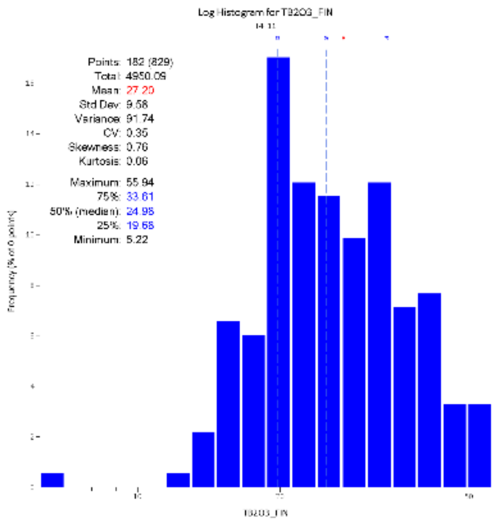


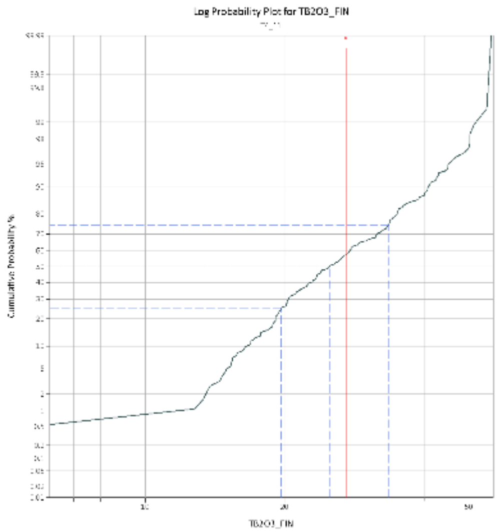
Pr2O3



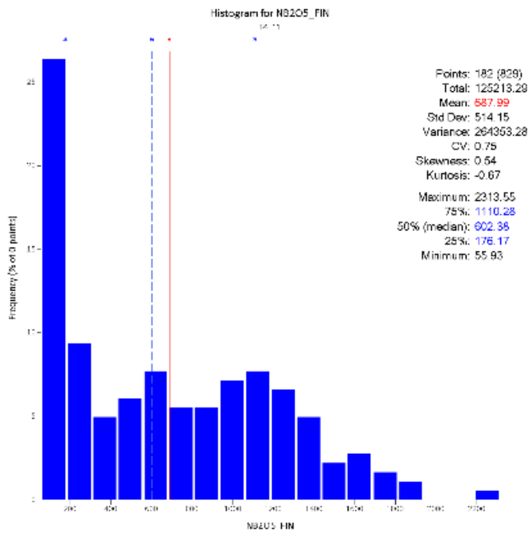


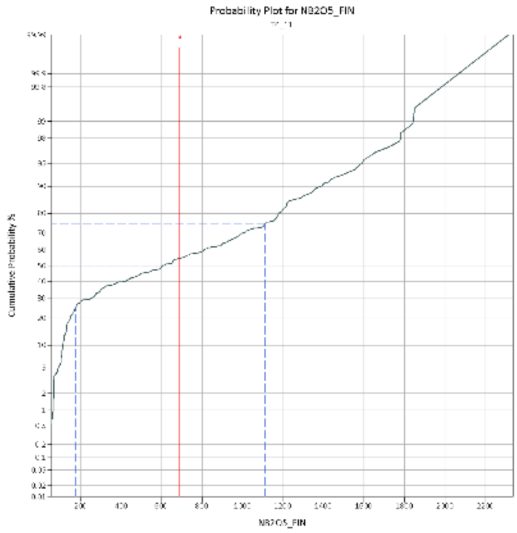
Tb2O3



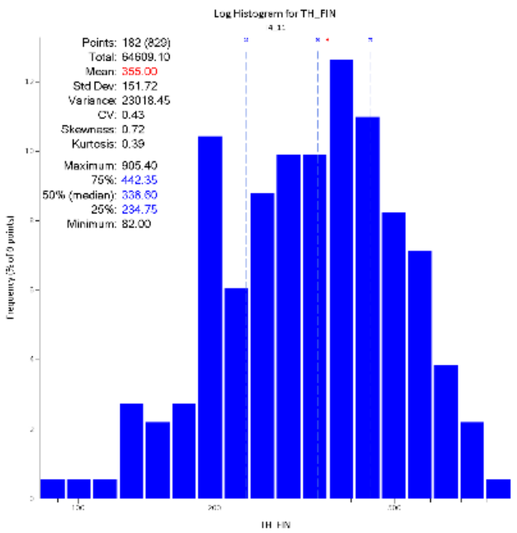


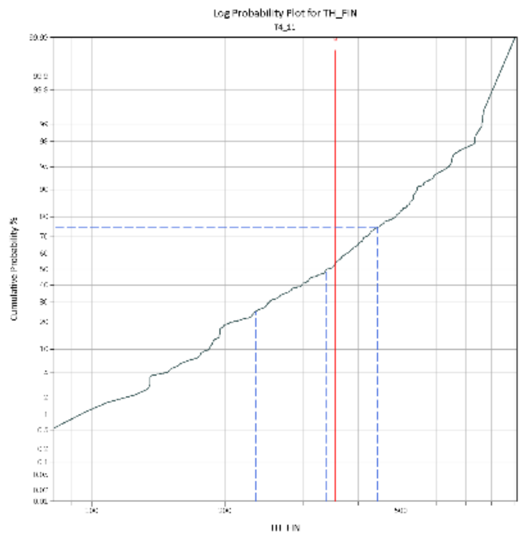
Nb2O5



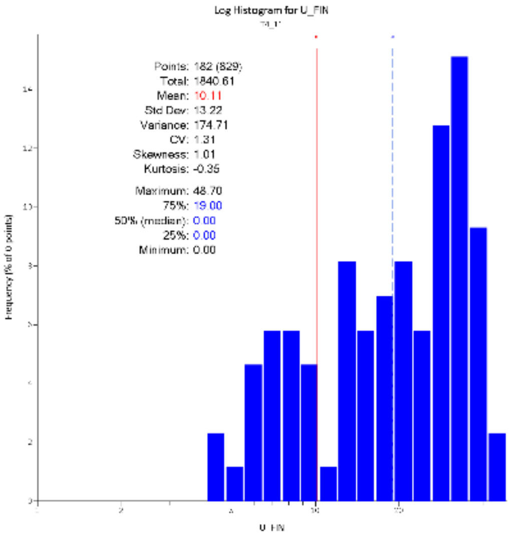


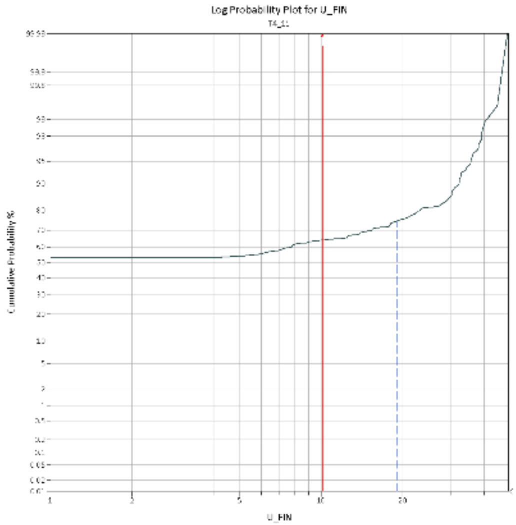
Th





U

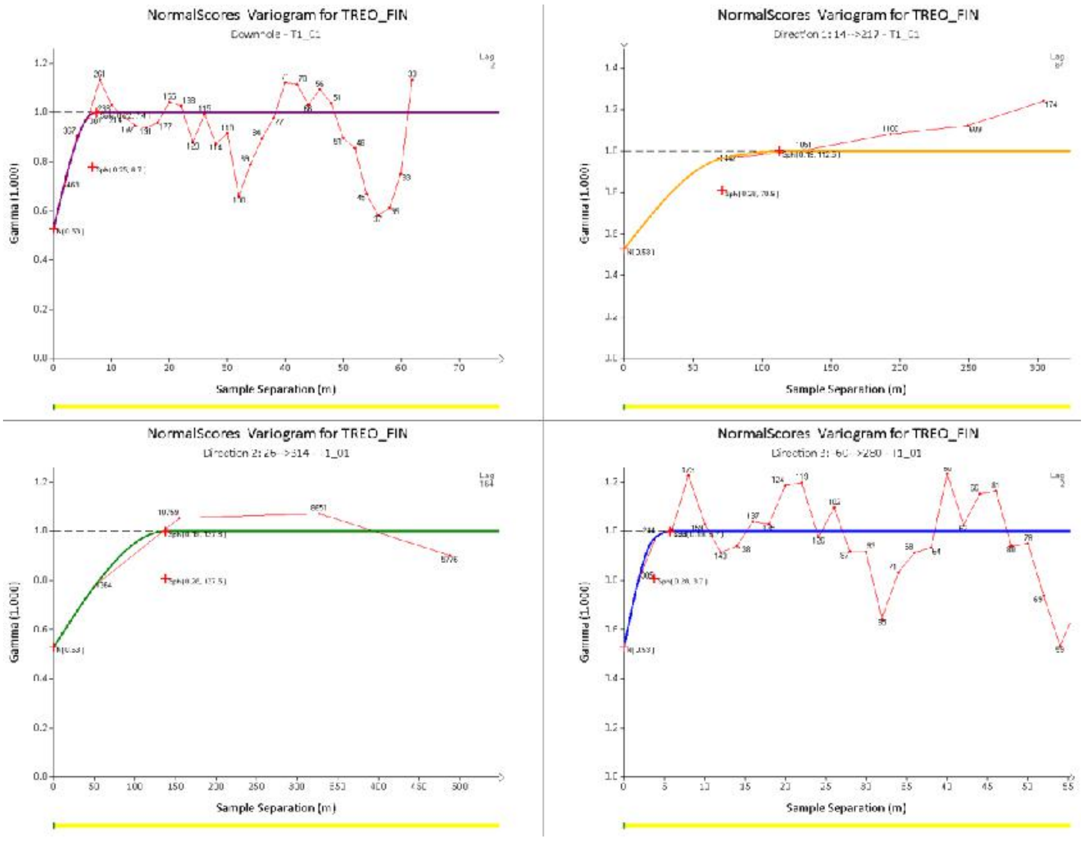

Log Histogram for U_FIN
Points: 182 (829)
Total: 1840.61
Mean: 10.11
Std Dev: 13.22
Variance: 174.71
CV: 1.31
Skewness: 1.01
Kurtosis: -0.35
Maximum: 48.70
75%: 19.00
50% (median): 0.00
25%: 0.00
Minimum: 0.00
U_FIN

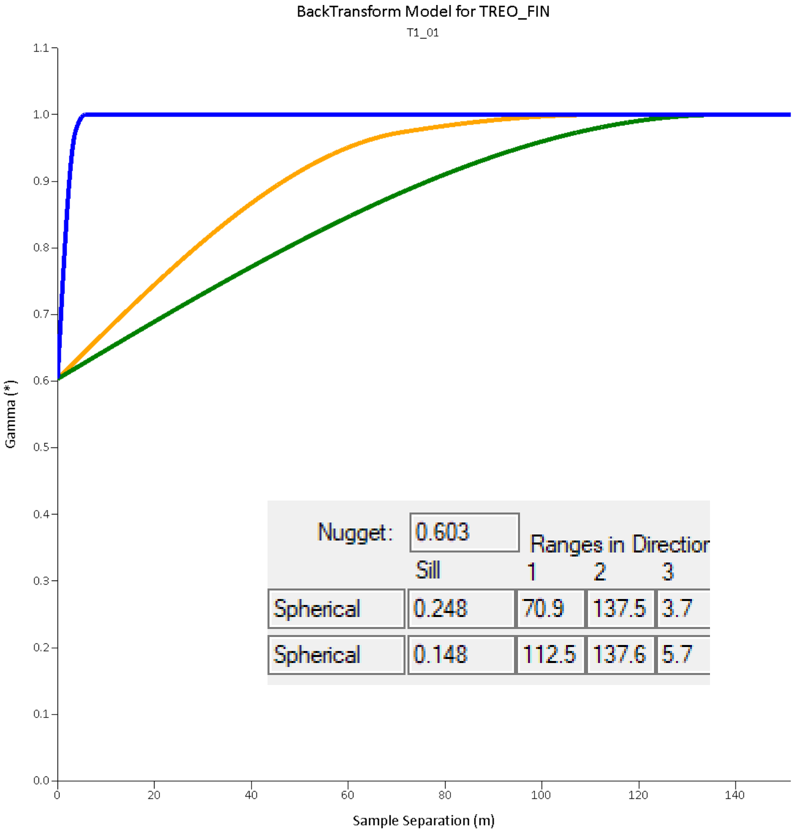

Log Probability Plot for U_FIN
U_FIN

Appendix E
Grade continuity models

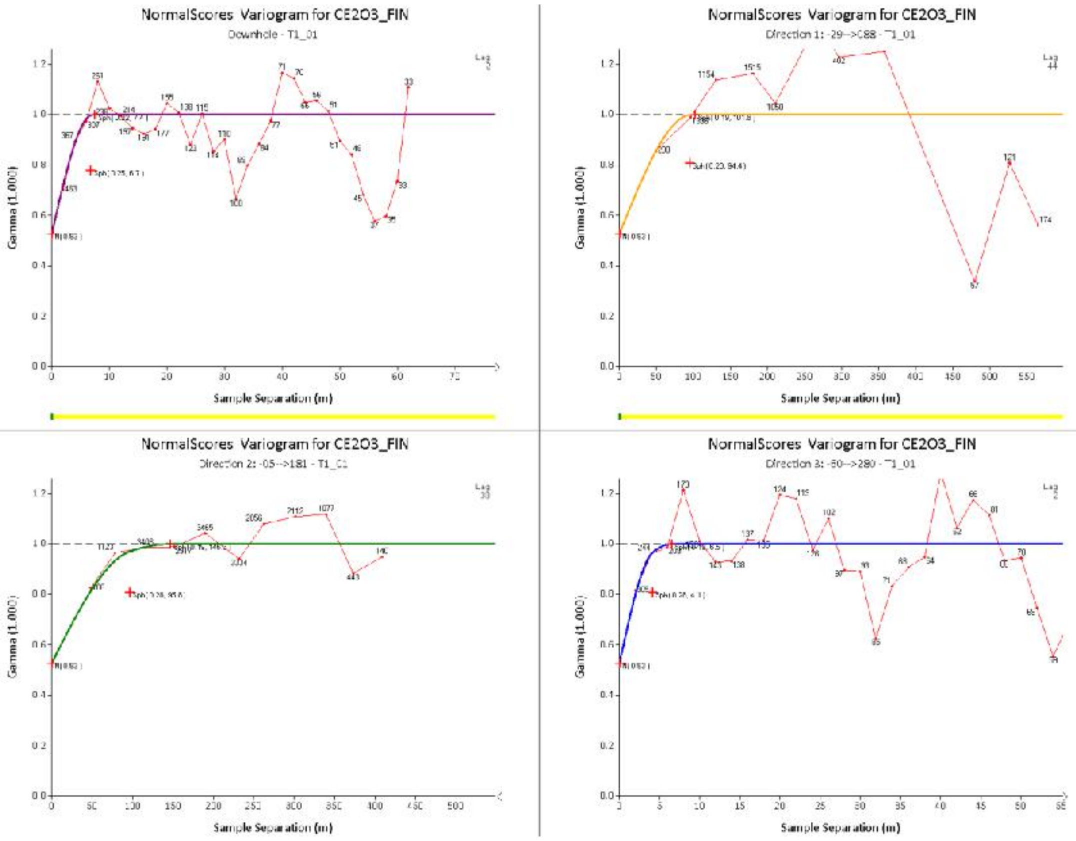
Target 1 Variogram models 0.5 to 1.0% TREO domain (ESTDOM T4_01)

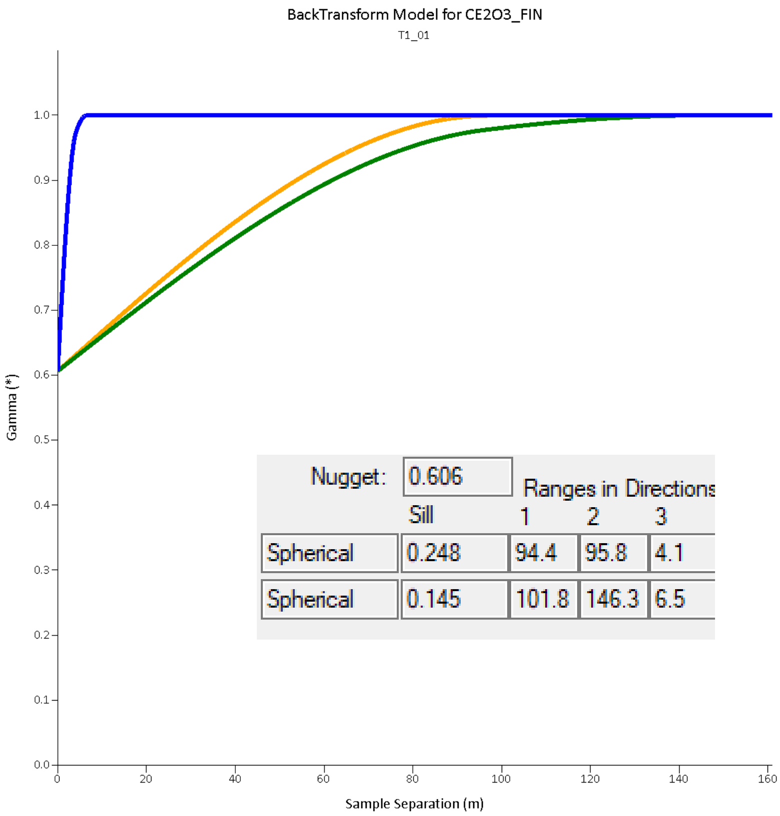
TREO



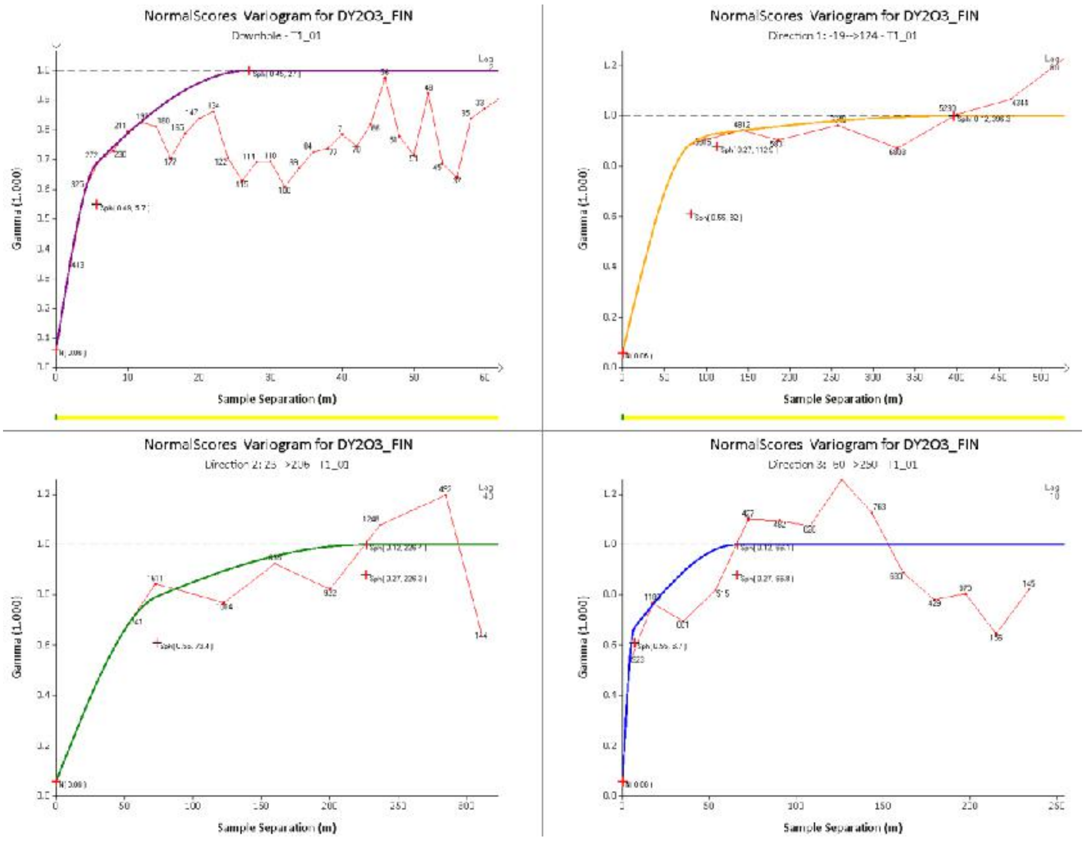


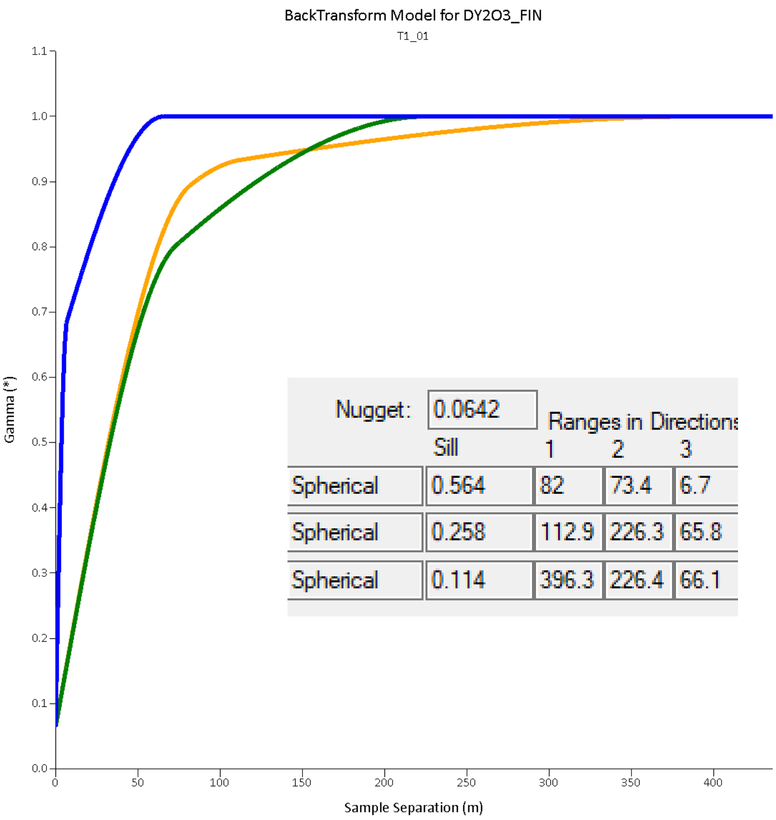
Ce_2O_3


NormalScores Variogram for CE2O3_FIN
Downhole - T1_01
NormalScores Variogram for CE2O3_FIN
Direction 1: -29-->088 - T1_01
NormalScores Variogram for CE2O3_FIN
Direction 2: -05-->181 - T1_01
NormalScores Variogram for CE2O3_FIN
Direction 3: -60-->280 - T1_01
Gamma (1.000)
Sample Separation (m)

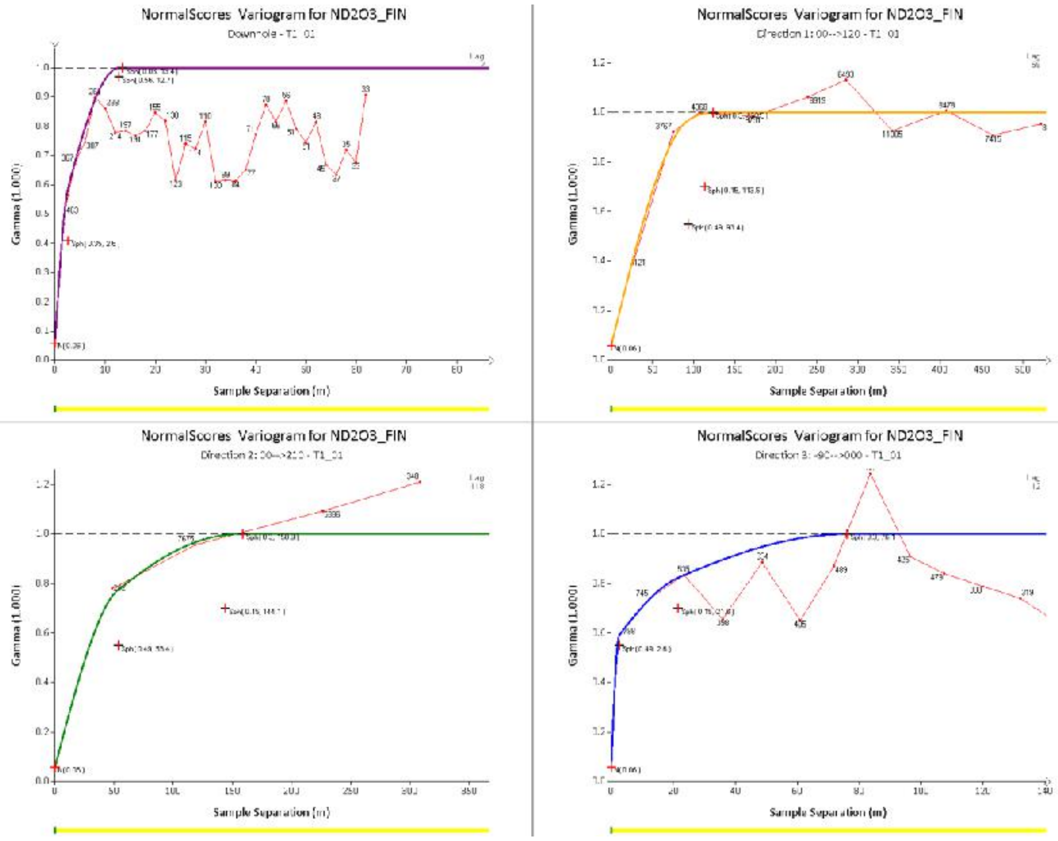

BackTransform Model for CE2O3_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.606
Ranges in Directions
Sill 1 2 3
Spherical 0.248 94.4 95.8 4.1
Spherical 0.145 101.8 146.3 6.5

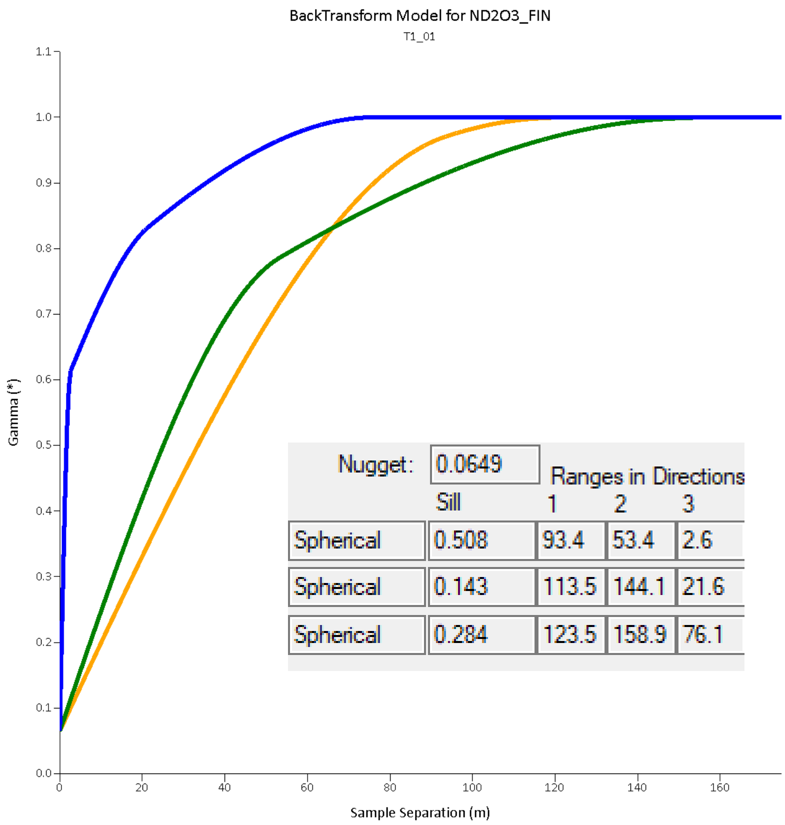
Dy_2O_3


NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)

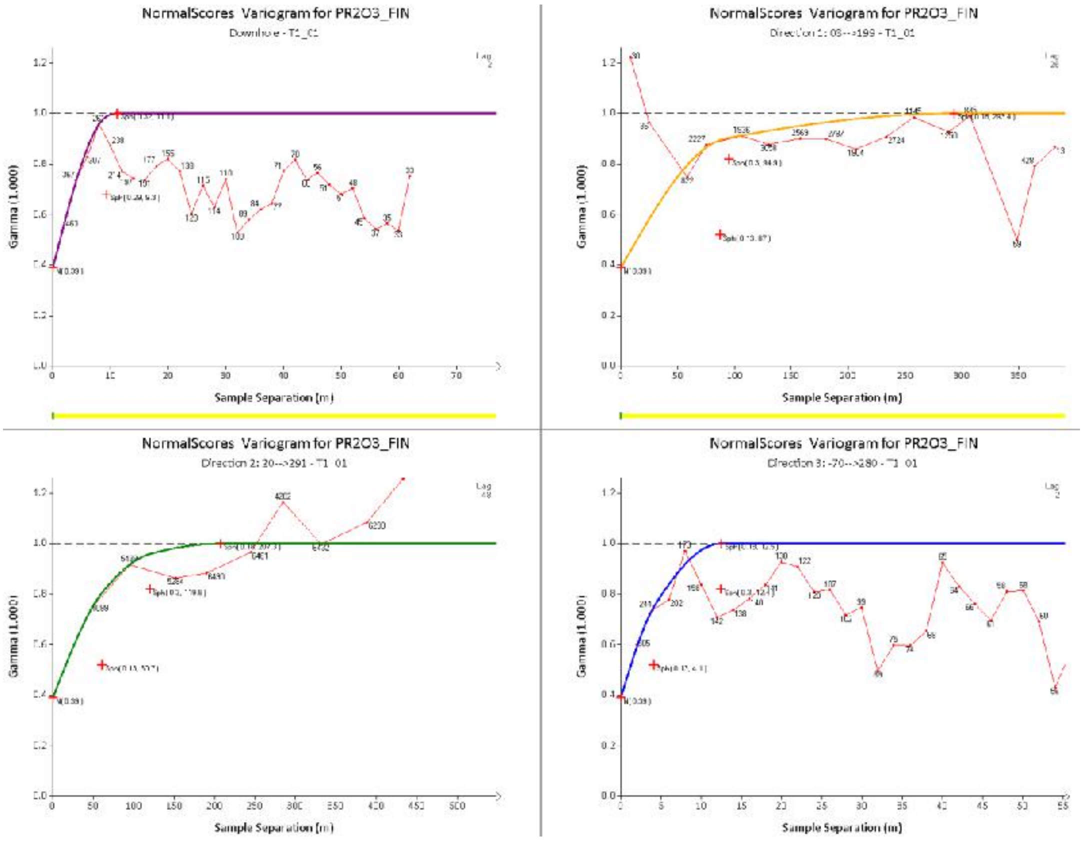

BackTransform Model for DY2O3_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.0642
Ranges in Directions
Sill 1 2 3
Spherical 0.564 82 73.4 6.7
Spherical 0.258 112.9 226.3 65.8
Spherical 0.114 396.3 226.4 66.1

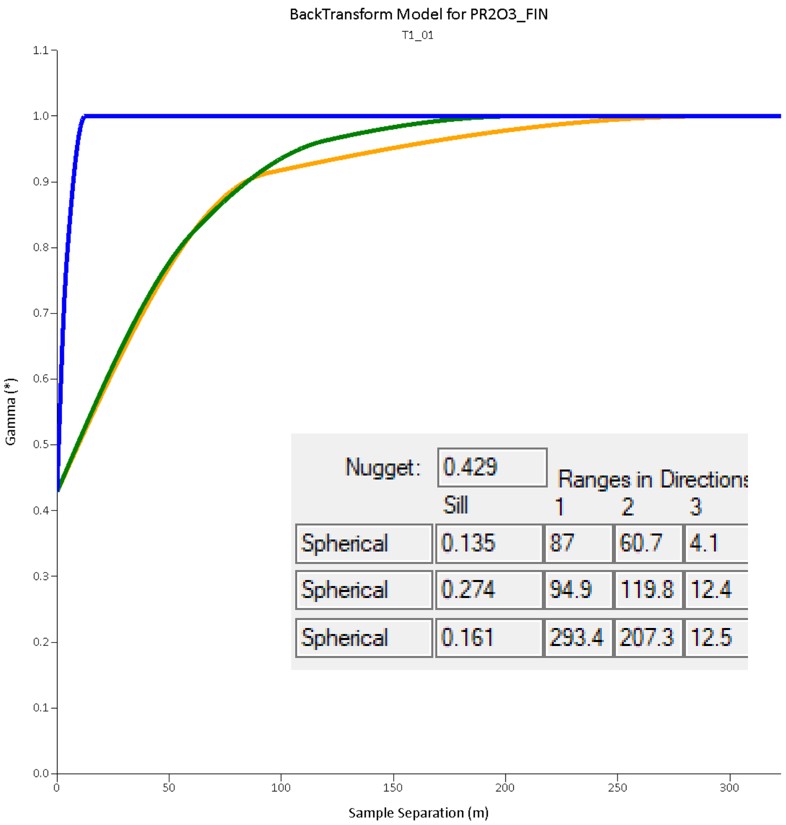
Nd_2O_3


NormalScores Variogram for ND2O3_FIN
Downhole - T1_01
NormalScores Variogram for ND2O3_FIN
Direction 1: 00-->120 - T1_01
NormalScores Variogram for ND2O3_FIN
NormalScores Variogram for ND2O3_FIN
Gamma (1.000)
Sample Separation (m)


BackTransform Model for ND2O3_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.0649
Ranges in Directions
Sill 1 2 3
Spherical 0.508 93.4 53.4 2.6
Spherical 0.143 113.5 144.1 21.6
Spherical 0.284 123.5 158.9 76.1

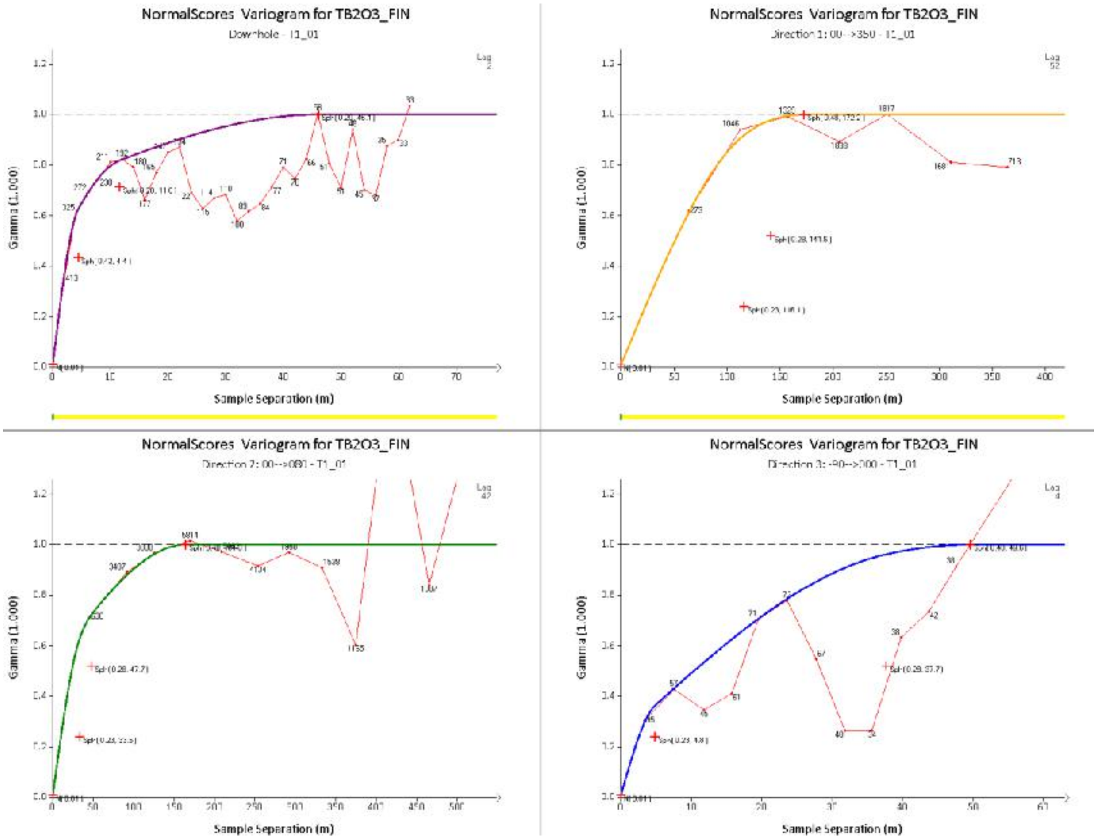
Pr_2O_3

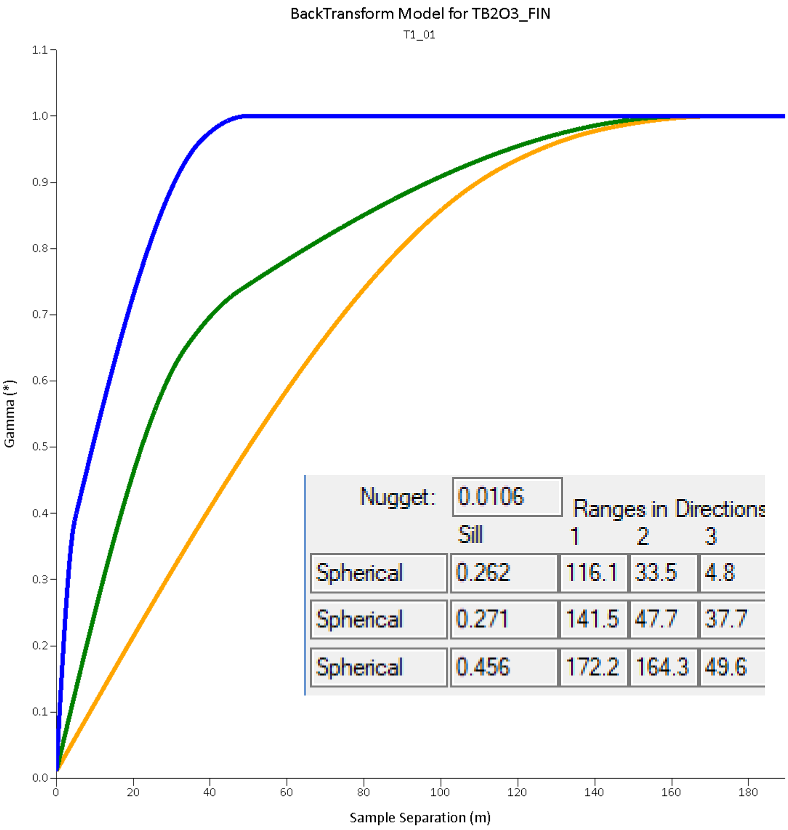




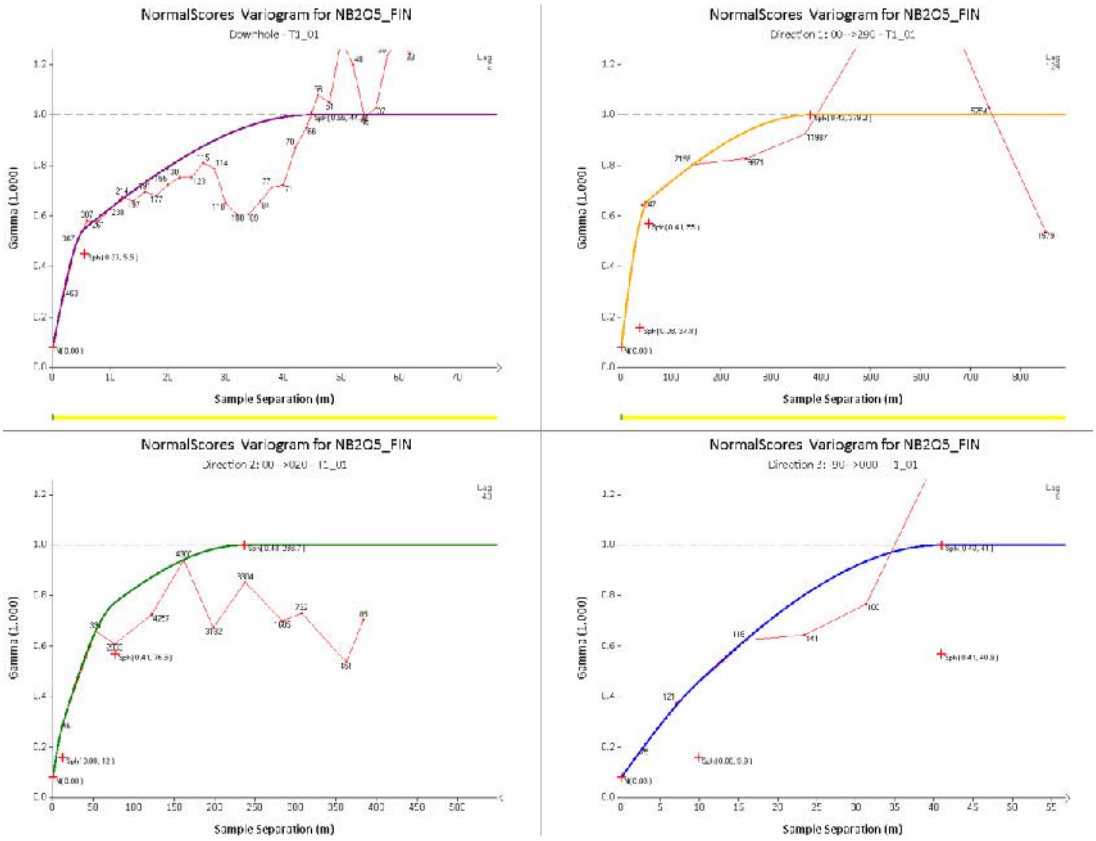
Nugget:	0.429	Ranges in Directions		
	Sill	1	2	3
Spherical	0.135	87	60.7	4.1
Spherical	0.274	94.9	119.8	12.4
Spherical	0.161	293.4	207.3	12.5

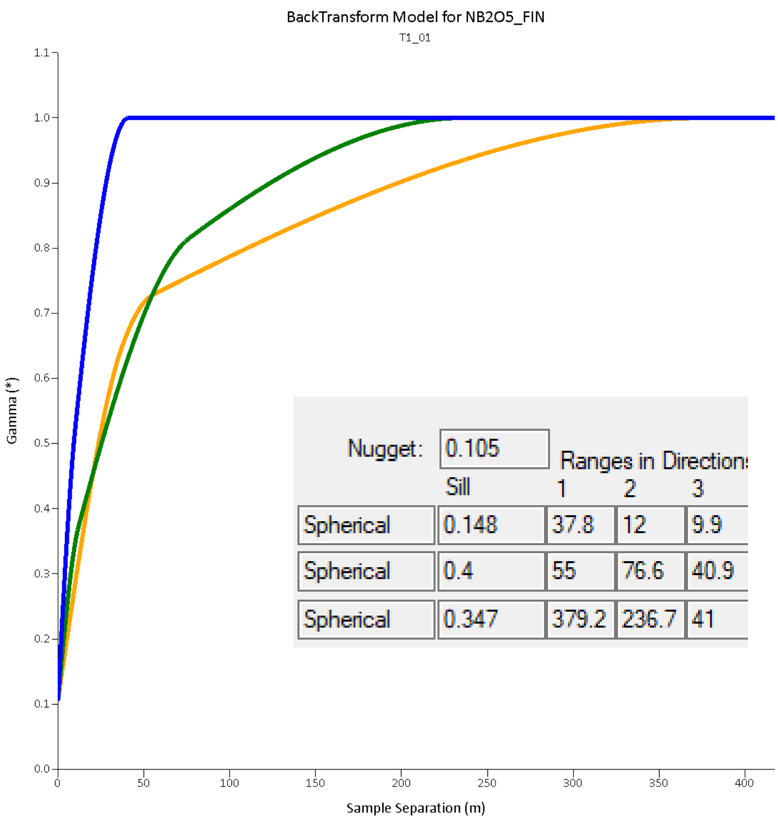
Tb_2O_3


NormalScores Variogram for TB2O3_FIN
Downhole - T1_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 1: 00-->350 - T1_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 2: 00-->080 - T1_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 3: -90-->000 - T1_01
Gamma (1.000)
Sample Separation (m)


BackTransform Model for TB2O3_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.0106
Ranges in Directions
Sill 1 2 3
Spherical 0.262 116.1 33.5 4.8
Spherical 0.271 141.5 47.7 37.7
Spherical 0.456 172.2 164.3 49.6

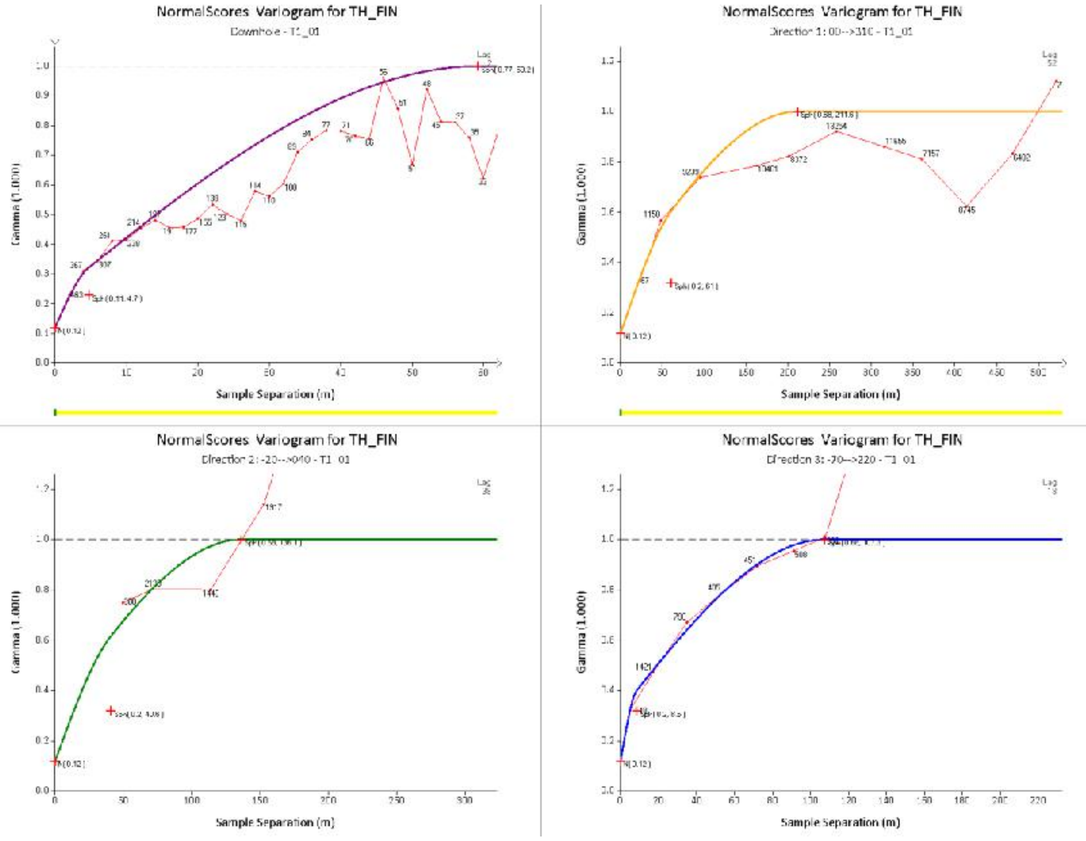
Nb_2O_5

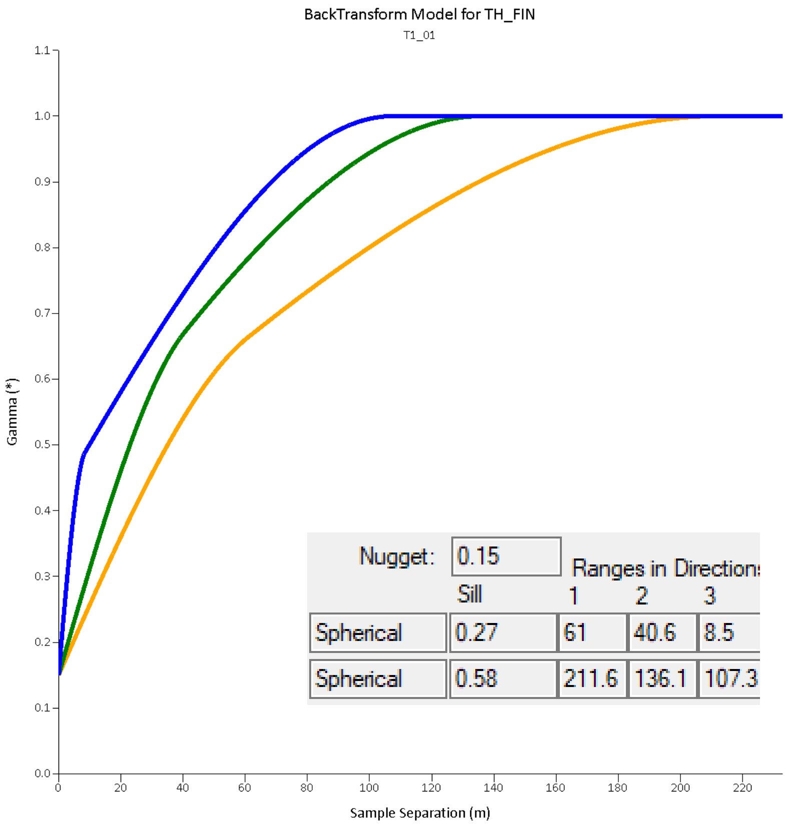




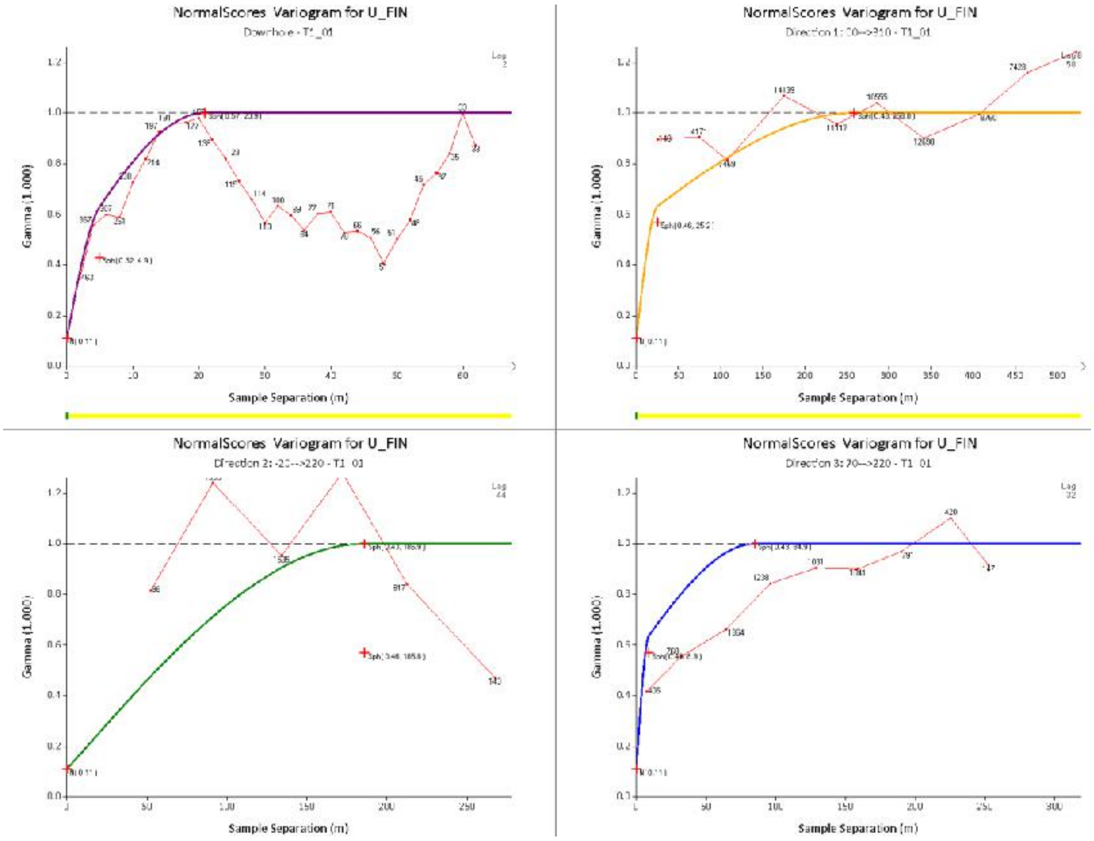
	Sill	1	2	3
Spherical	0.148	37.8	12	9.9
Spherical	0.4	55	76.6	40.9
Spherical	0.347	379.2	236.7	41

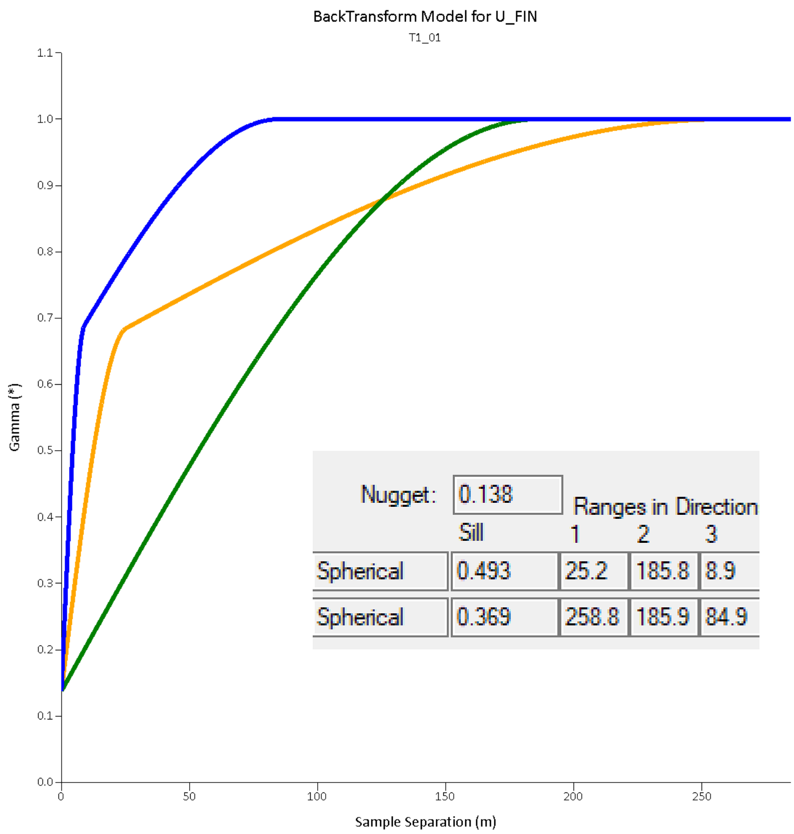
Th


NormalScores Variogram for TH_FIN
Downhole - T1_01
NormalScores Variogram for TH_FIN
Direction 1: 00-->310 - T1_01
NormalScores Variogram for TH_FIN
Direction 2: -20-->040 - T1_01
NormalScores Variogram for TH_FIN
Direction 3: -70-->220 - T1_01
Gamma (1.000)
Sample Separation (m)


BackTransform Model for TH_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.15
Ranges in Directions
Sill 1 2 3
Spherical 0.27 61 40.6 8.5
Spherical 0.58 211.6 136.1 107.3

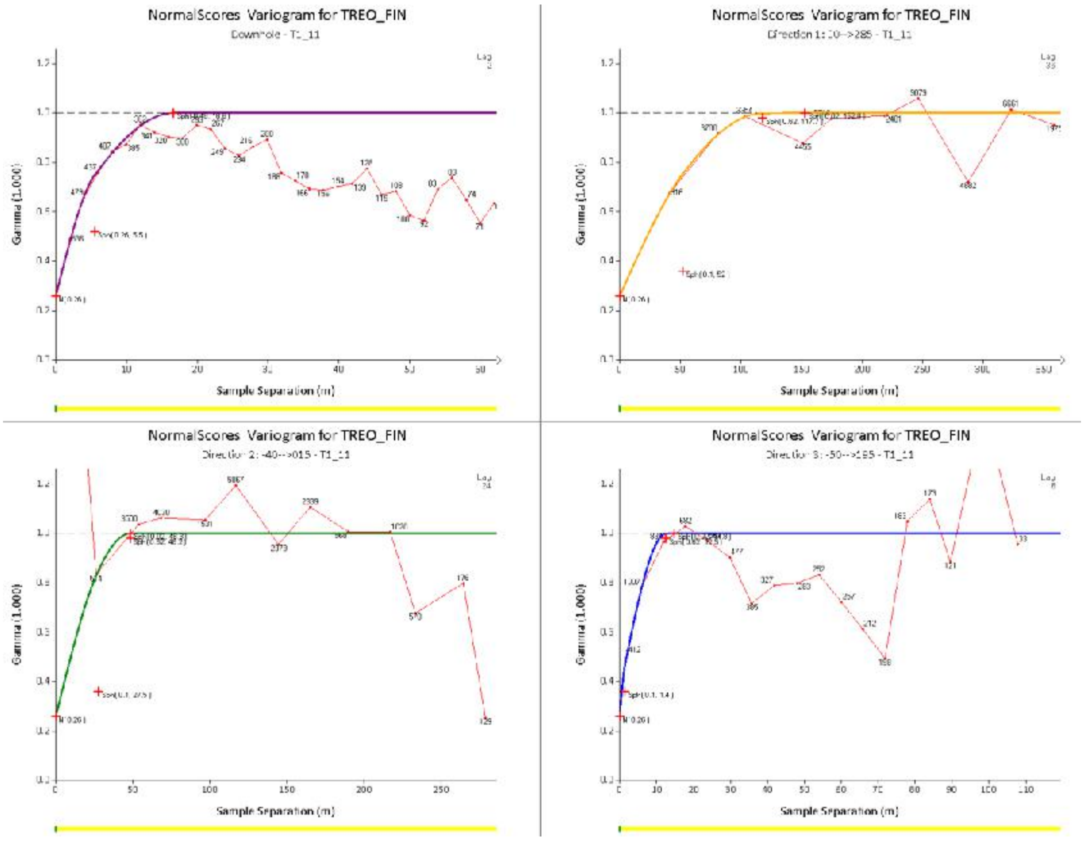
U

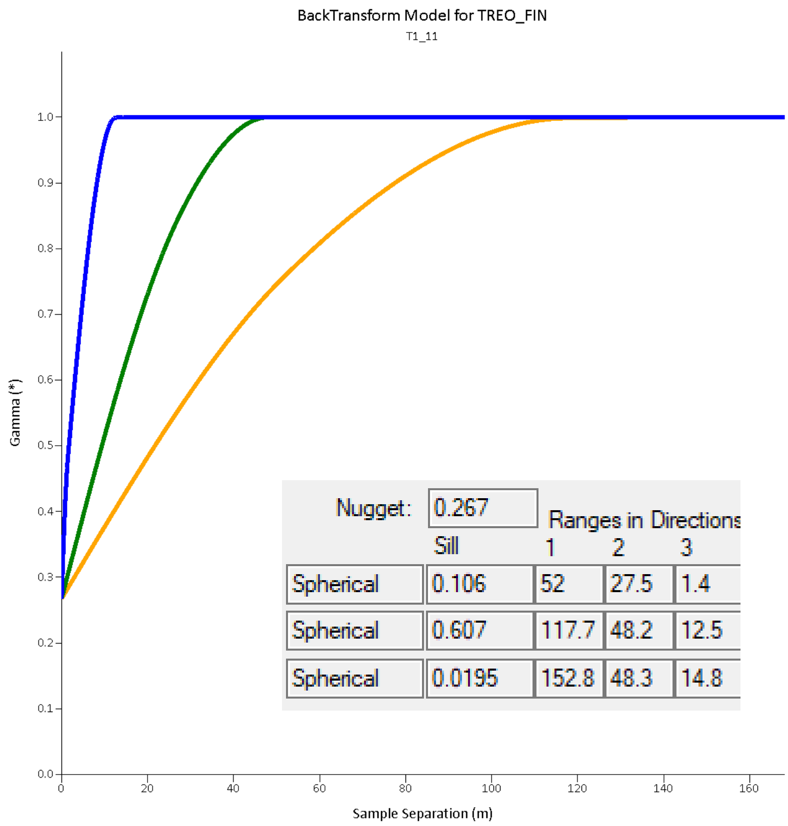

NormalScores Variogram for U_FIN
Downhole - T1_01
NormalScores Variogram for U_FIN
Direction 1: 00-->310 - T1_01
NormalScores Variogram for U_FIN
Direction 2: -20-->220 - T1_01
NormalScores Variogram for U_FIN
Direction 3: 70-->220 - T1_01
Gamma (1.000)
Sample Separation (m)


BackTransform Model for U_FIN
T1_01
Gamma (*)
Sample Separation (m)
Nugget: 0.138
Ranges in Direction
Sill 1 2 3
Spherical 0.493 25.2 185.8 8.9
Spherical 0.369 258.8 185.9 84.9

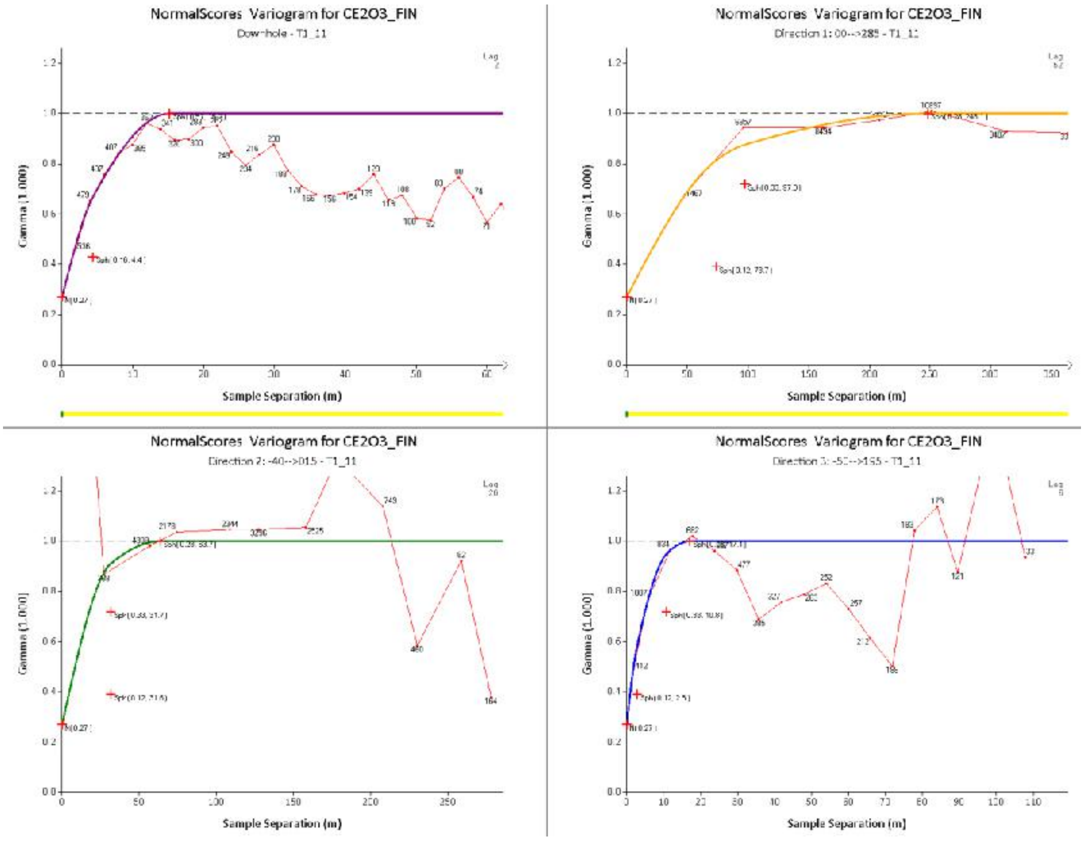
Target 1 Variogram models ≥1.0% TREO domain (ESTDOM T4_11)

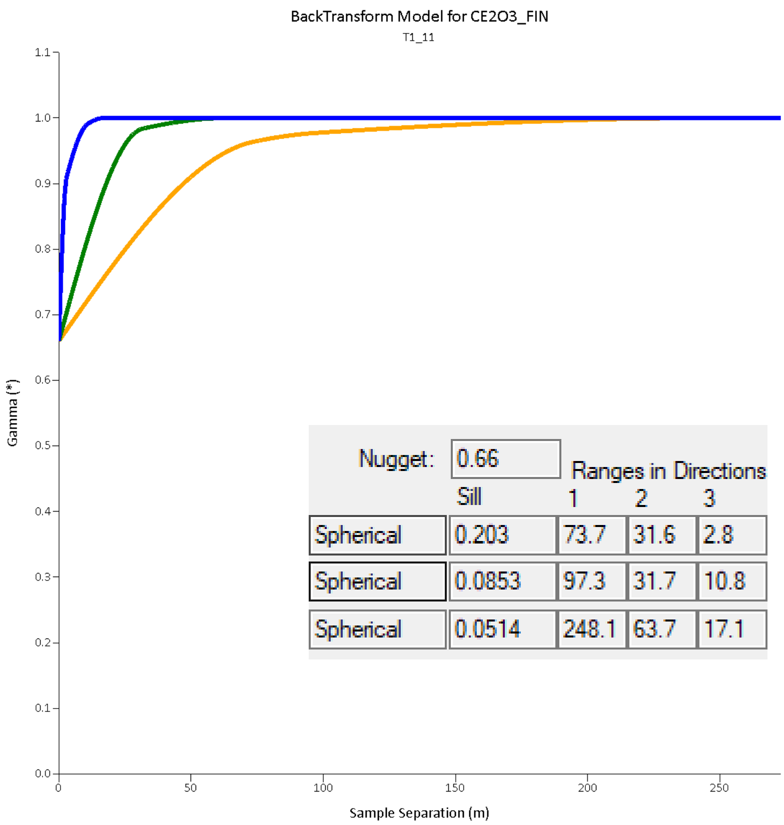
TREO



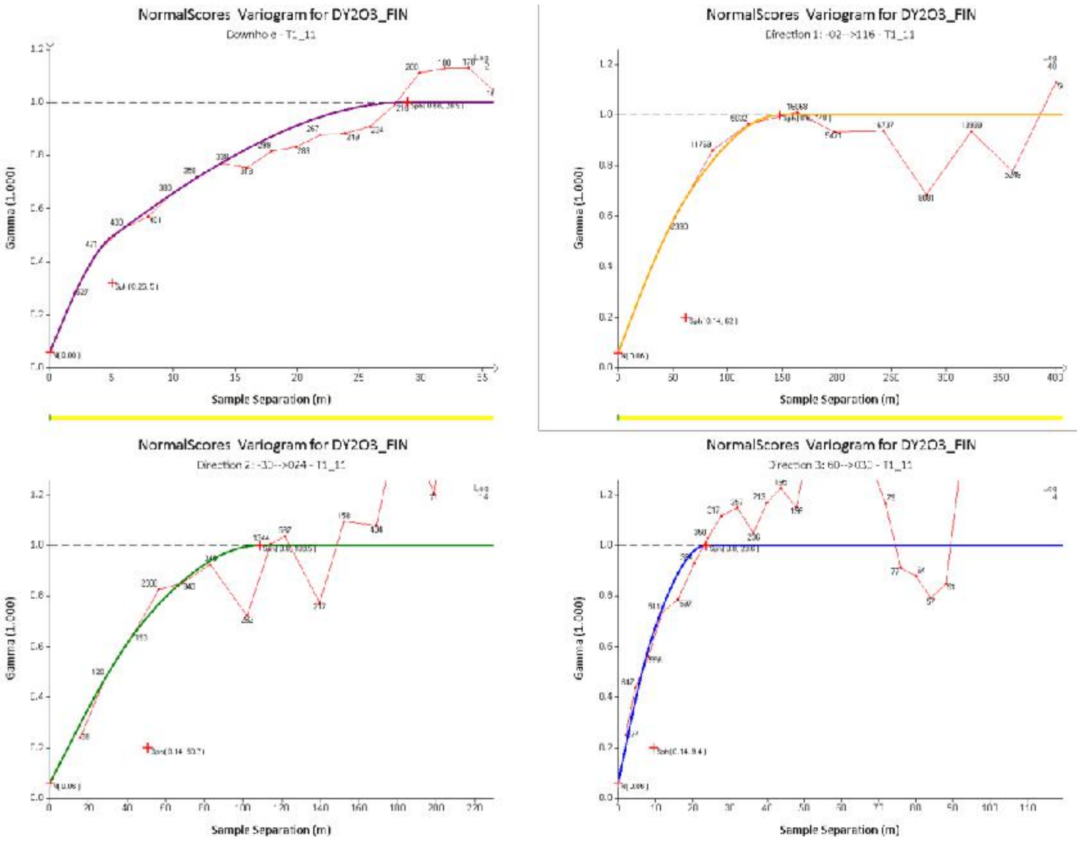


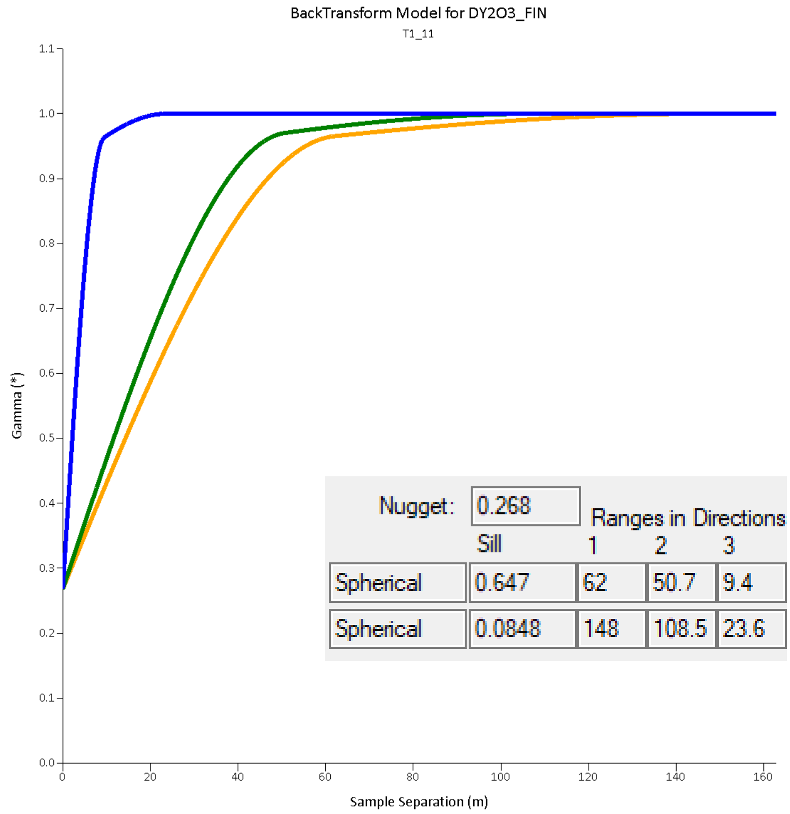
Ce_2O_3


NormalScores Variogram for CE2O3_FIN
Downhole - T1_11
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Direction 1: 00-->285 - T1_11
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Direction 2: -40-->015 - T1_11
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Direction 3: -50-->195 - T1_11
Sample Separation (m)

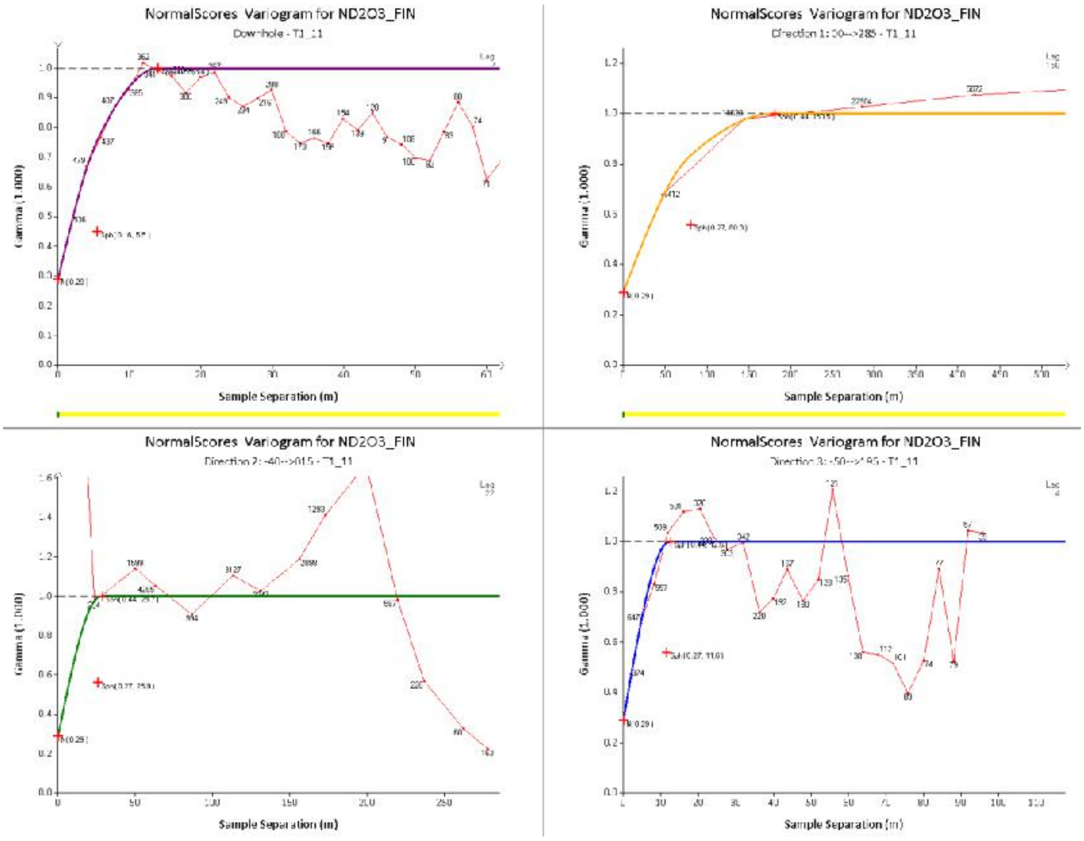

BackTransform Model for CE2O3_FIN
T1_11
Gamma (*)
Nugget: 0.66
Ranges in Directions
Sill 1 2 3
Spherical 0.203 73.7 31.6 2.8
Spherical 0.0853 97.3 31.7 10.8
Spherical 0.0514 248.1 63.7 17.1
Sample Separation (m)

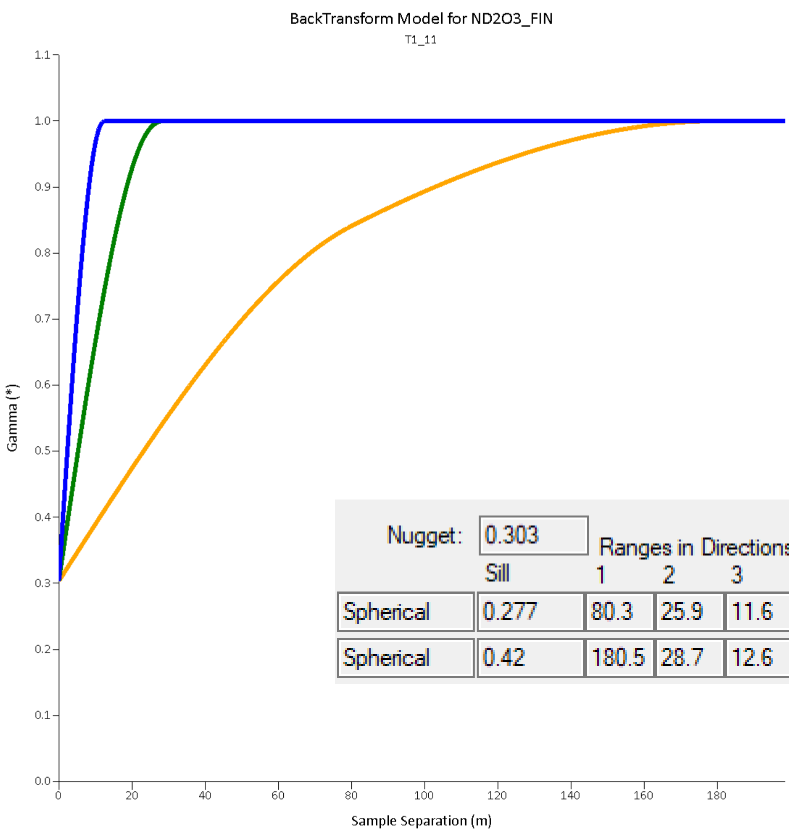
Dy_2O_3


NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)

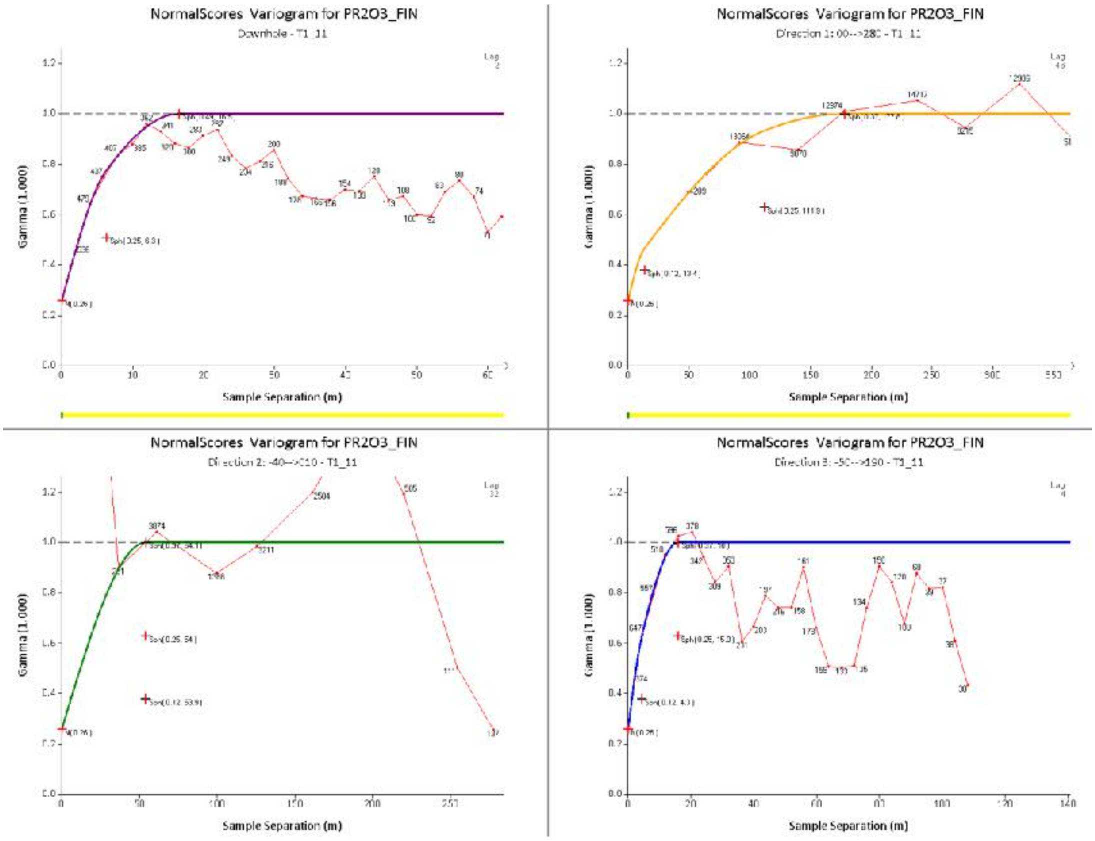

BackTransform Model for DY2O3_FIN
T1_11
Gamma (*)
Sample Separation (m)
Nugget: 0.268
Ranges in Directions
Sill 1 2 3
Spherical 0.647 62 50.7 9.4
Spherical 0.0848 148 108.5 23.6

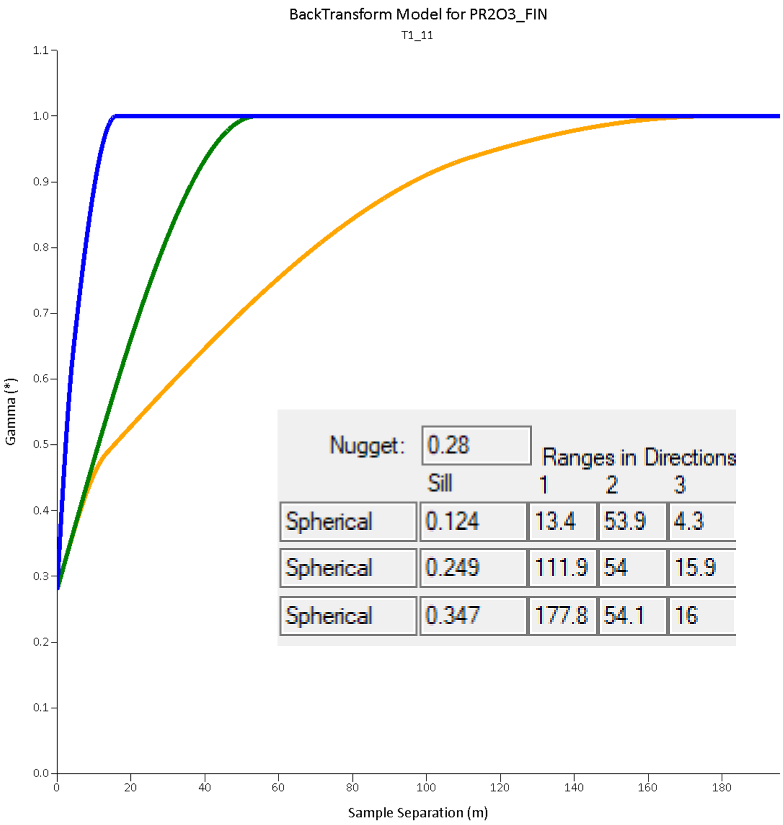
Nd_2O_3


NormalScores Variogram for ND2O3_FIN
Sample Separation (m)
Gamma (1.000)


BackTransform Model for ND2O3_FIN
T1_11
Gamma (*)
Sample Separation (m)
Nugget: 0.303
Ranges in Directions
Sill 1 2 3
Spherical 0.277 80.3 25.9 11.6
Spherical 0.42 180.5 28.7 12.6

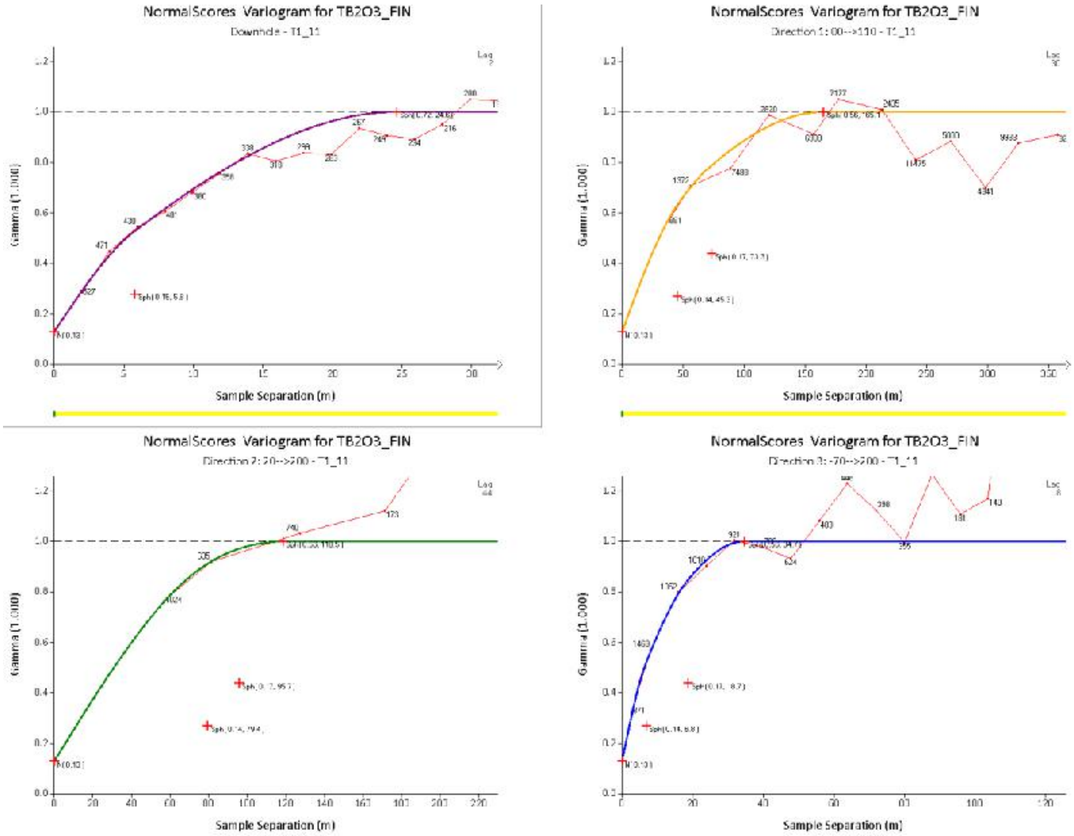
Pr_2O_3

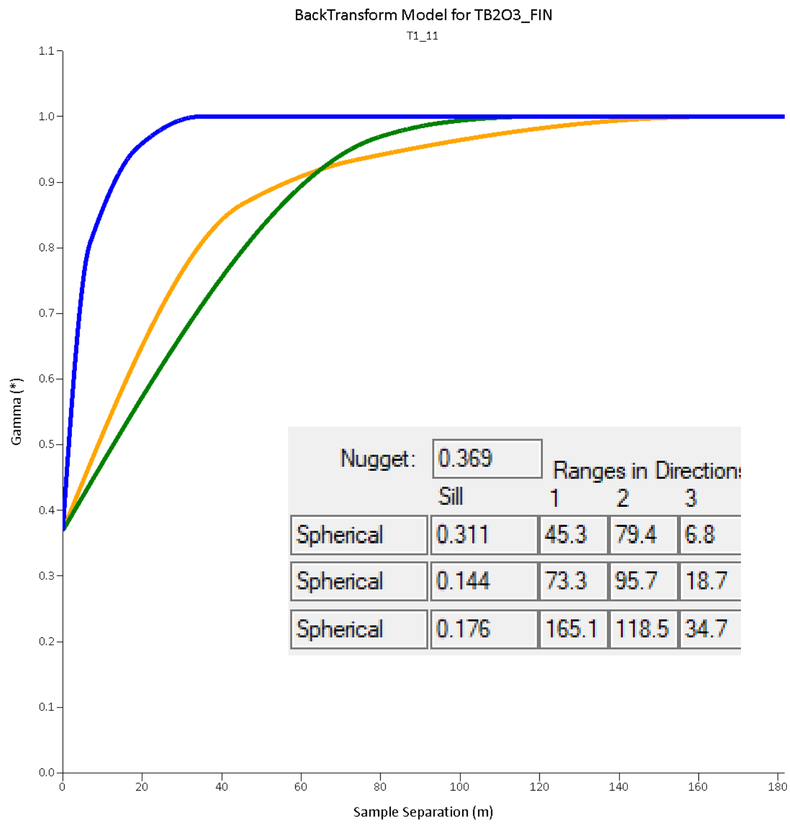




	Sill	1	2	3
Spherical	0.124	13.4	53.9	4.3
Spherical	0.249	111.9	54	15.9
Spherical	0.347	177.8	54.1	16

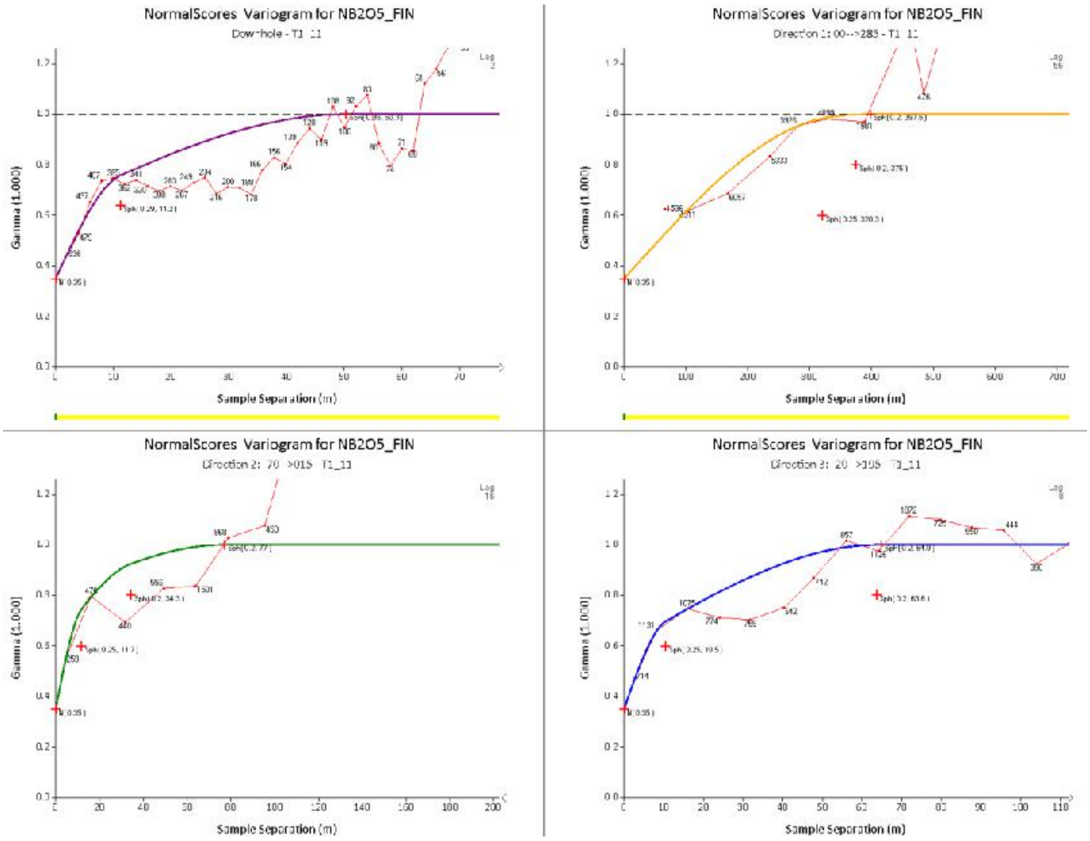
Tb_2O_3

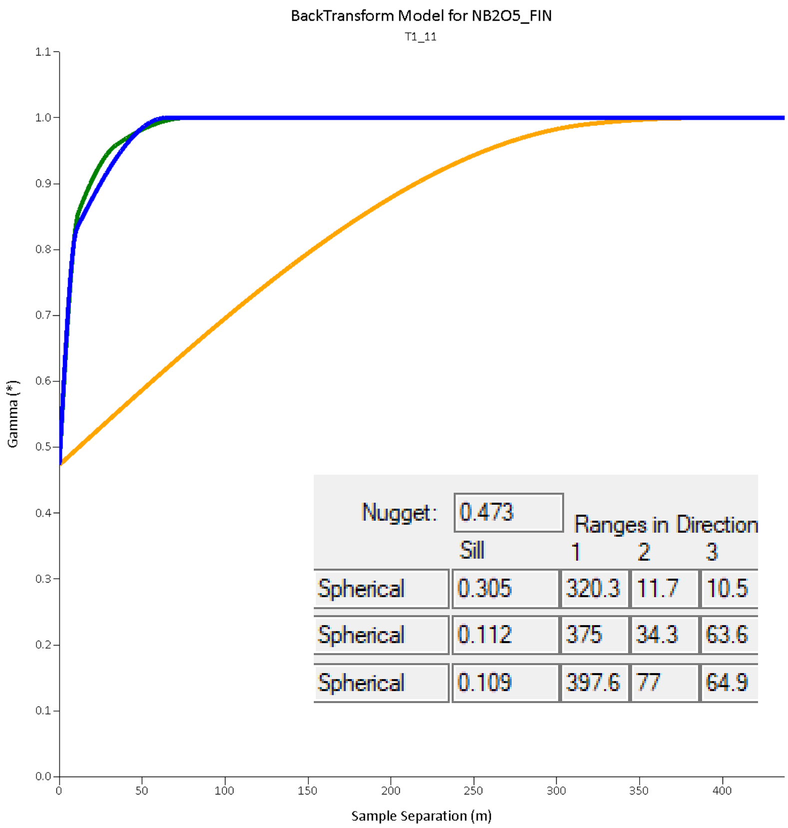




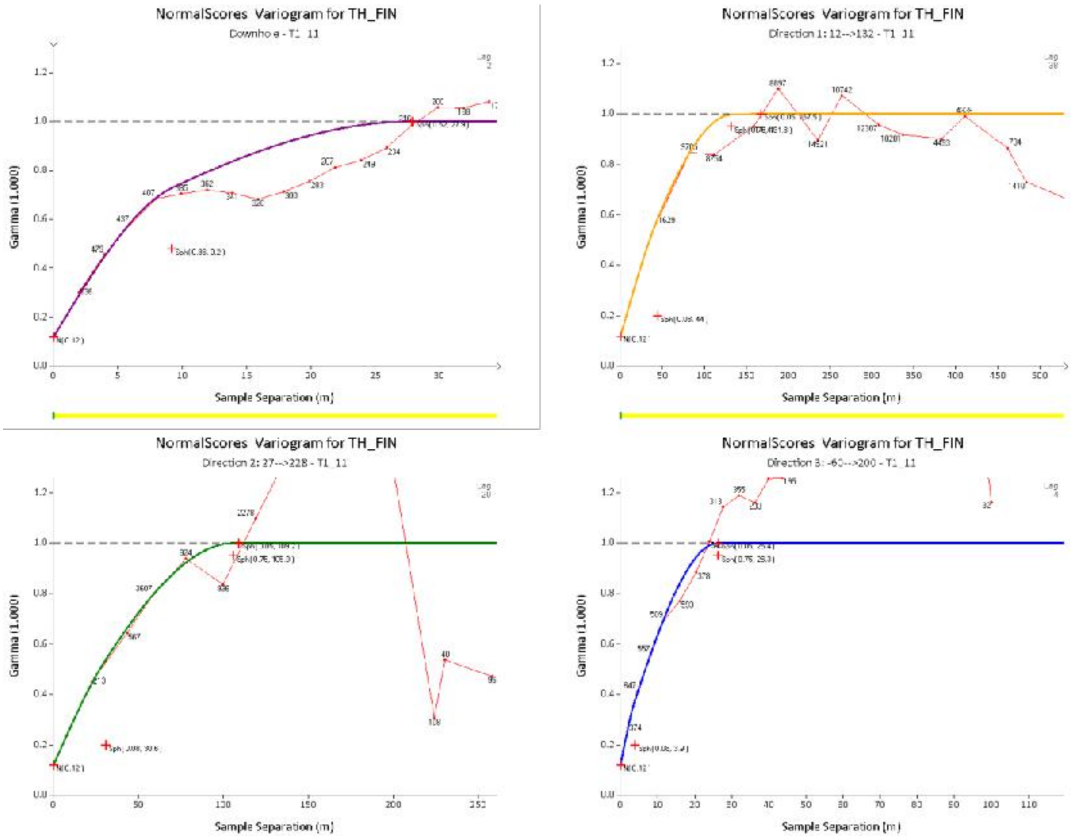
	Sill	1	2	3
Spherical	0.311	45.3	79.4	6.8
Spherical	0.144	73.3	95.7	18.7
Spherical	0.176	165.1	118.5	34.7

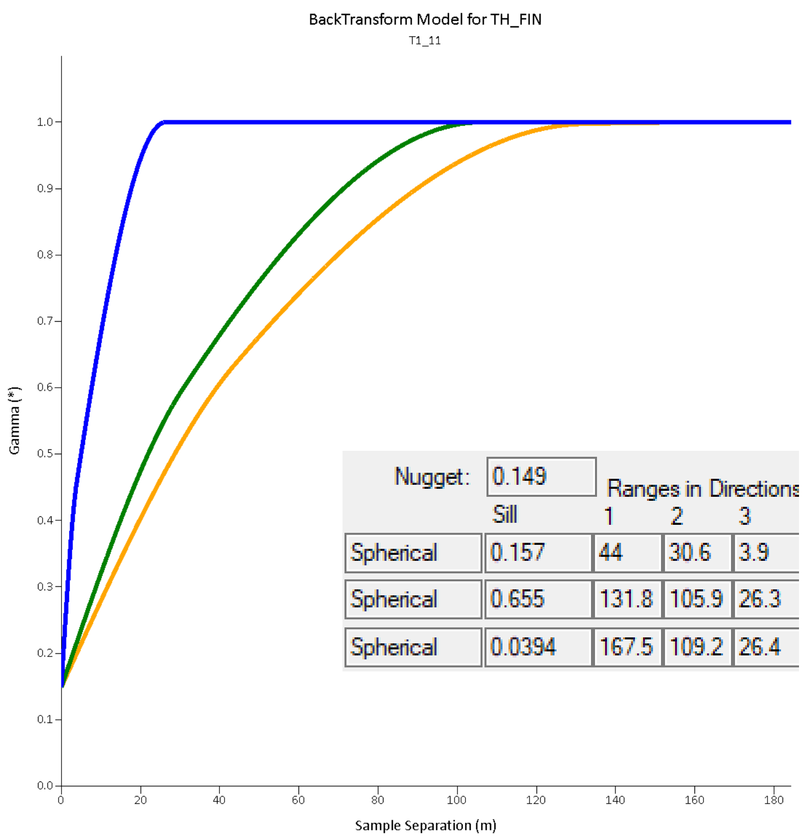
Nb_2O_5


NormalScores Variogram for NB2O5_FIN
Down-hole - T1_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 1: 00-->283 - T1_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 2: 70 >015 T1_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 3: 20 >195 T1_11
Gamma (1.000)
Sample Separation (m)


BackTransform Model for NB2O5_FIN
T1_11
Gamma (*)
Sample Separation (m)
Nugget: 0.473
Ranges in Direction
Sill 1 2 3
Spherical 0.305 320.3 11.7 10.5
Spherical 0.112 375 34.3 63.6
Spherical 0.109 397.6 77 64.9

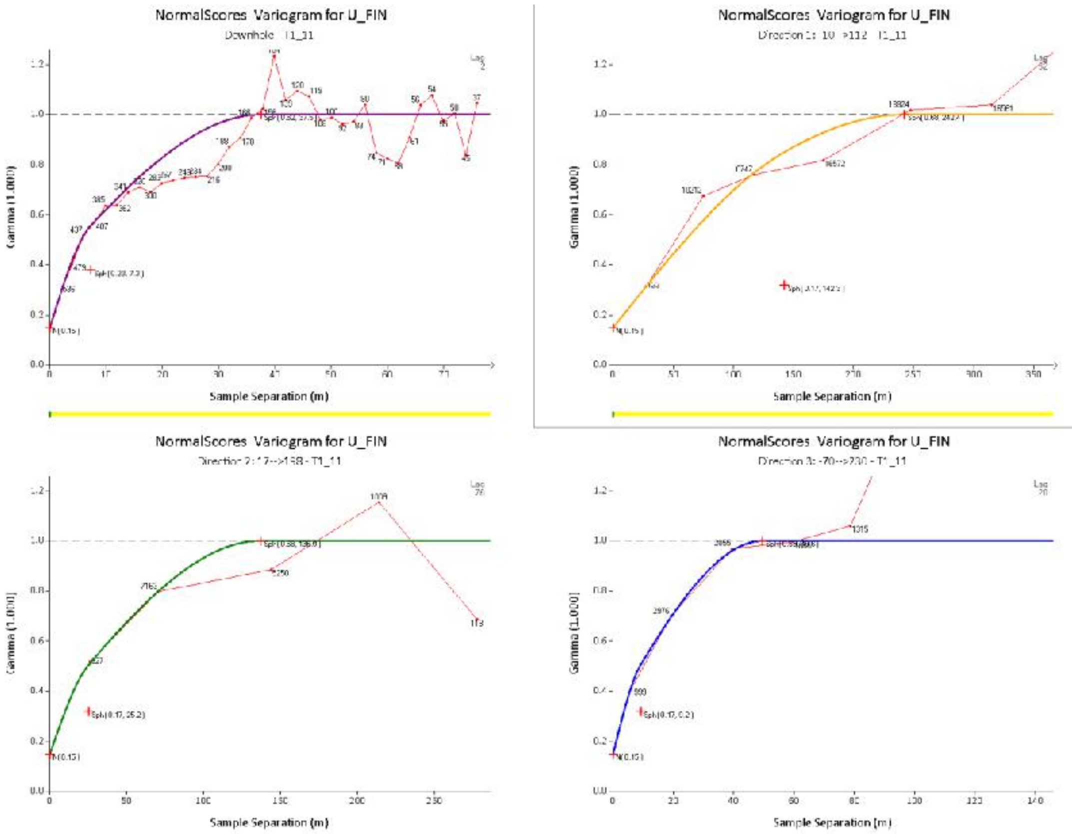
Th

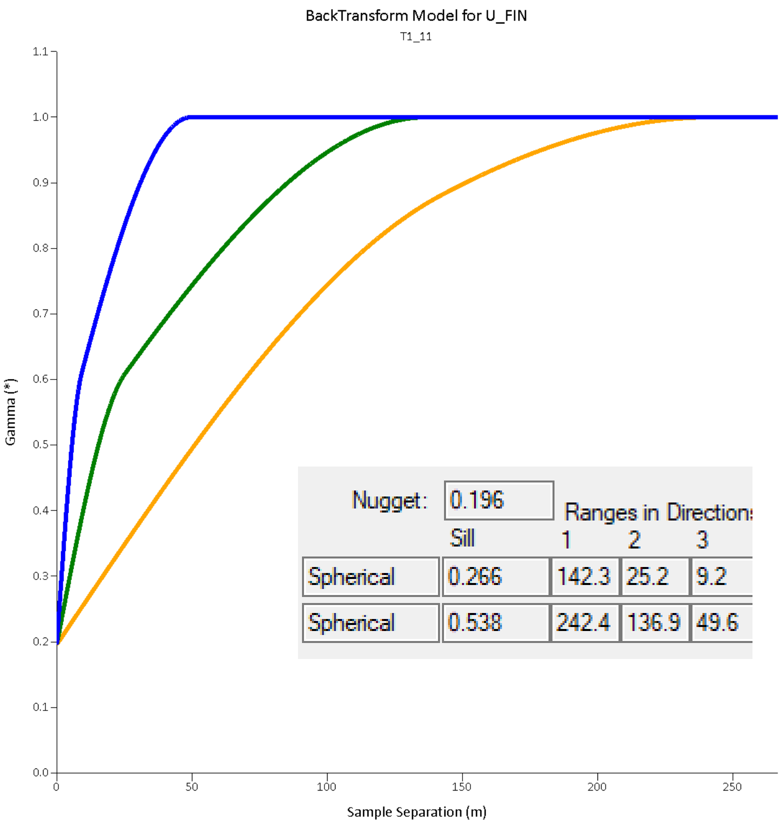




Nugget:	0.149	Ranges in Directions		
	Sill	1	2	3
Spherical	0.157	44	30.6	3.9
Spherical	0.655	131.8	105.9	26.3
Spherical	0.0394	167.5	109.2	26.4

U

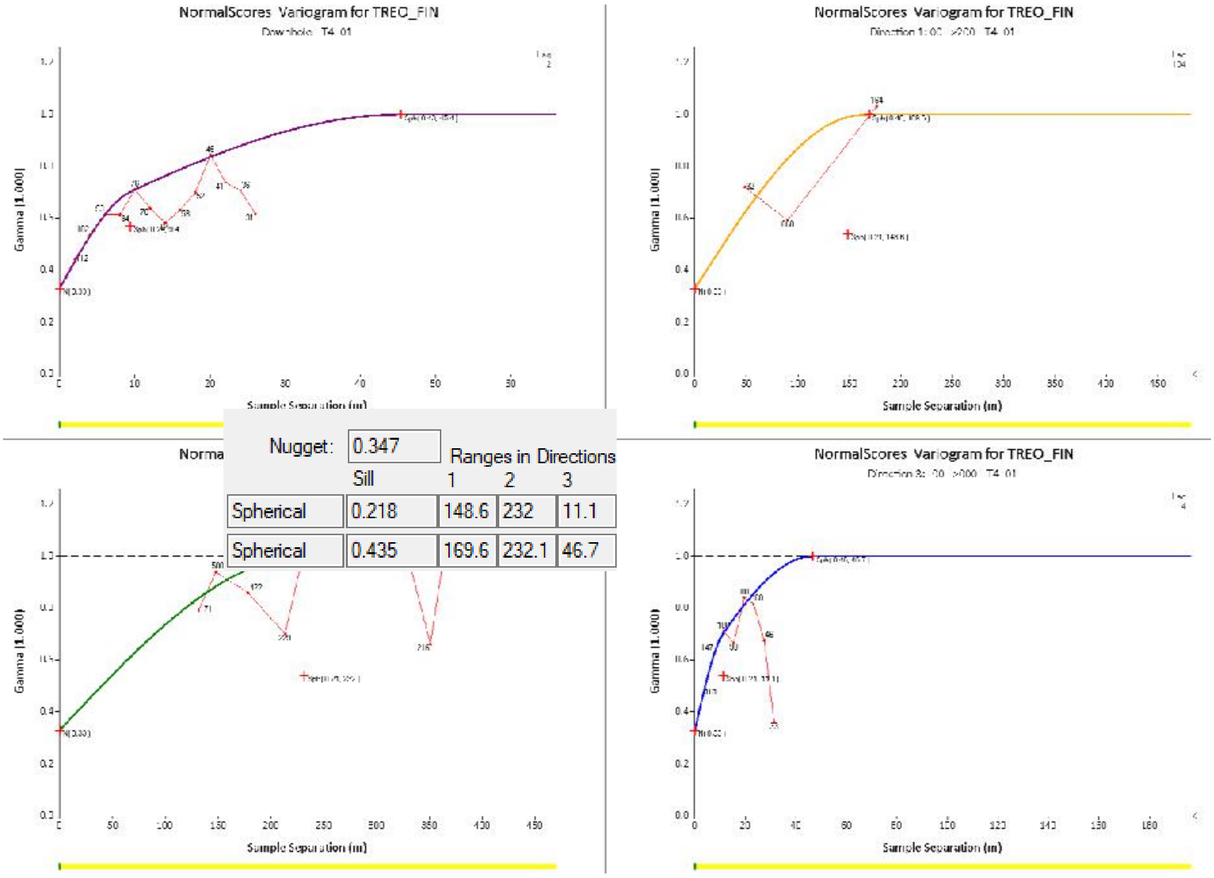




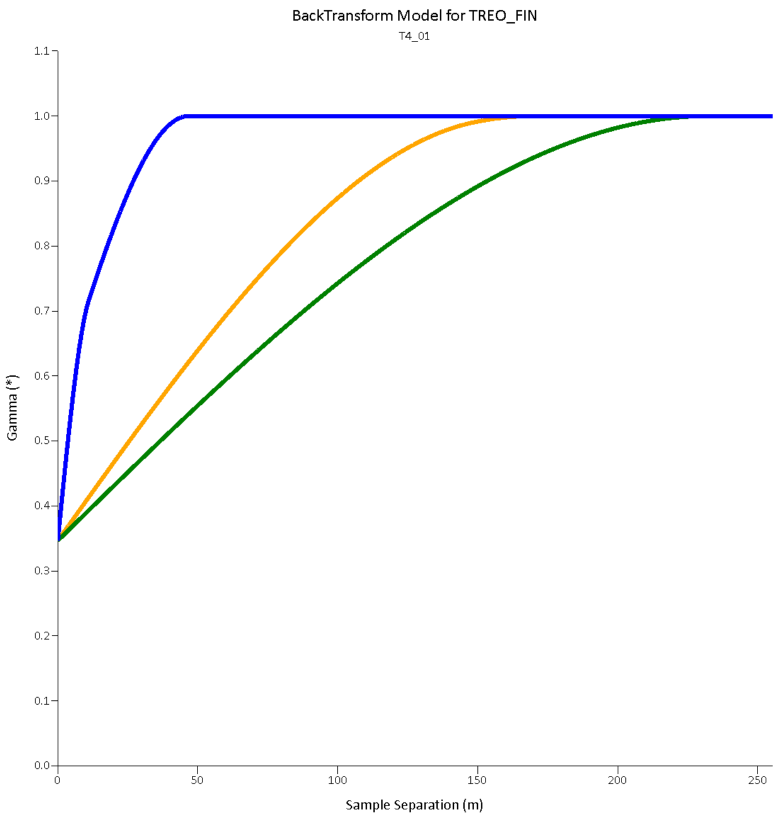
		Ranges in Directions		
Nugget: 0.196	Sill	1	2	3
Spherical	0.266	142.3	25.2	9.2
Spherical	0.538	242.4	136.9	49.6

Target 1 Variogram models 0.5 - 1.0% TREO domain (ESTDOM T4_01)

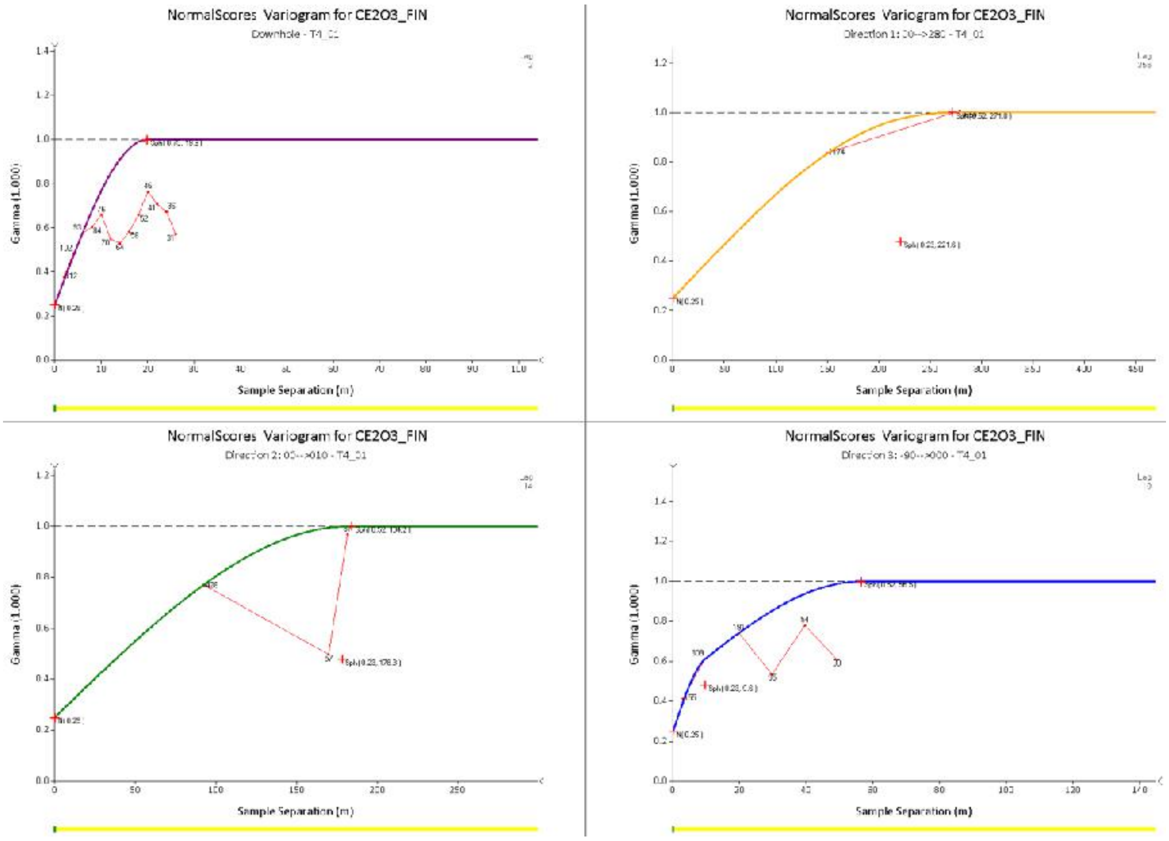
TREO

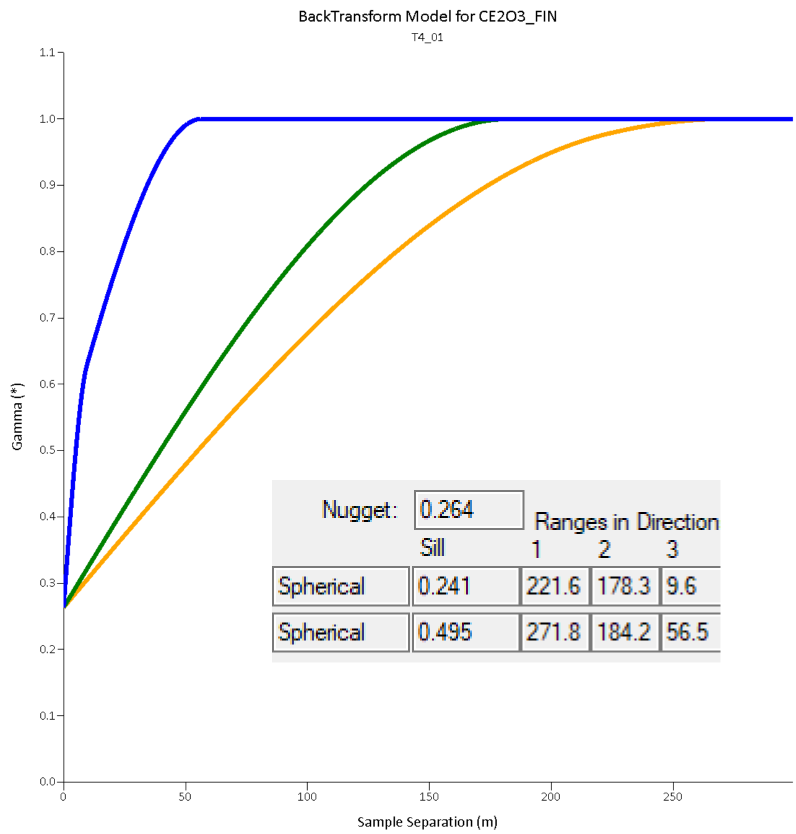


Nugget:	0.347	Ranges in Directions		
	Sill	1	2	3
Spherical	0.218	148.6	232	11.1
Spherical	0.435	169.6	232.1	46.7

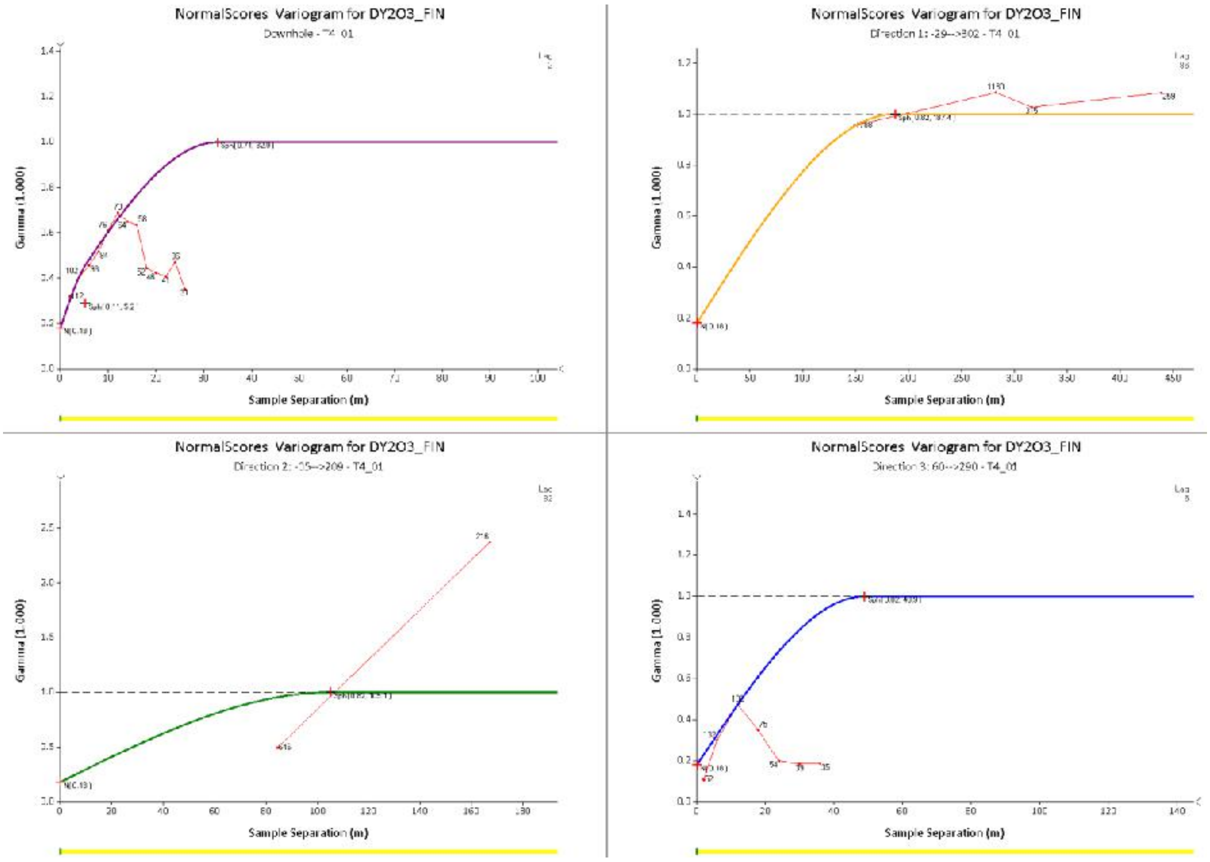


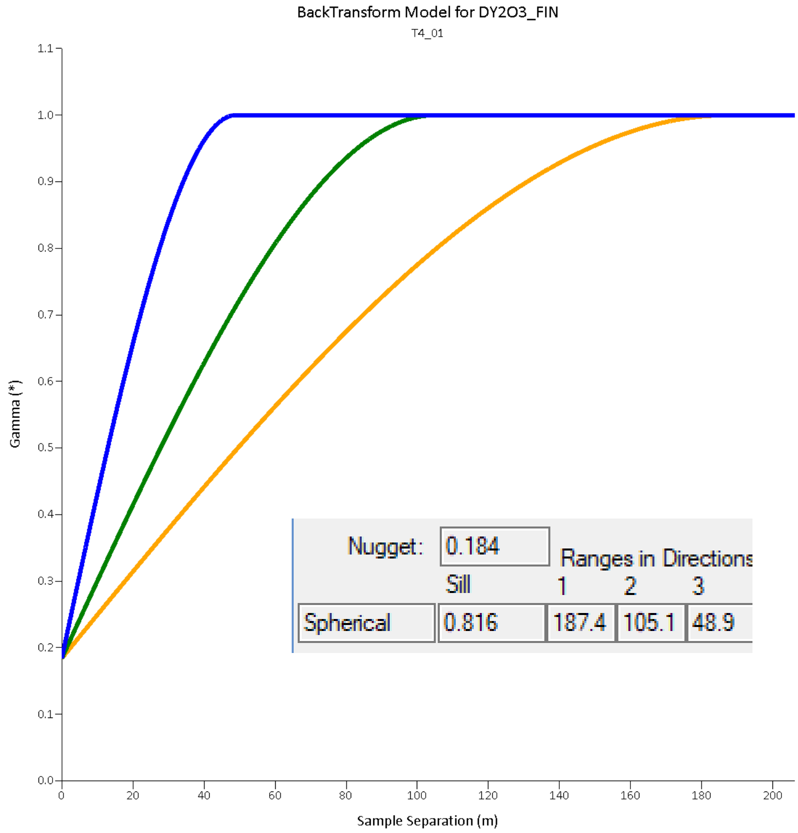
Ce_2O_3


NormalScores Variogram for CE2O3_FIN
Downhole - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for CE2O3_FIN
Gamma (1.000)
Sample Separation (m)

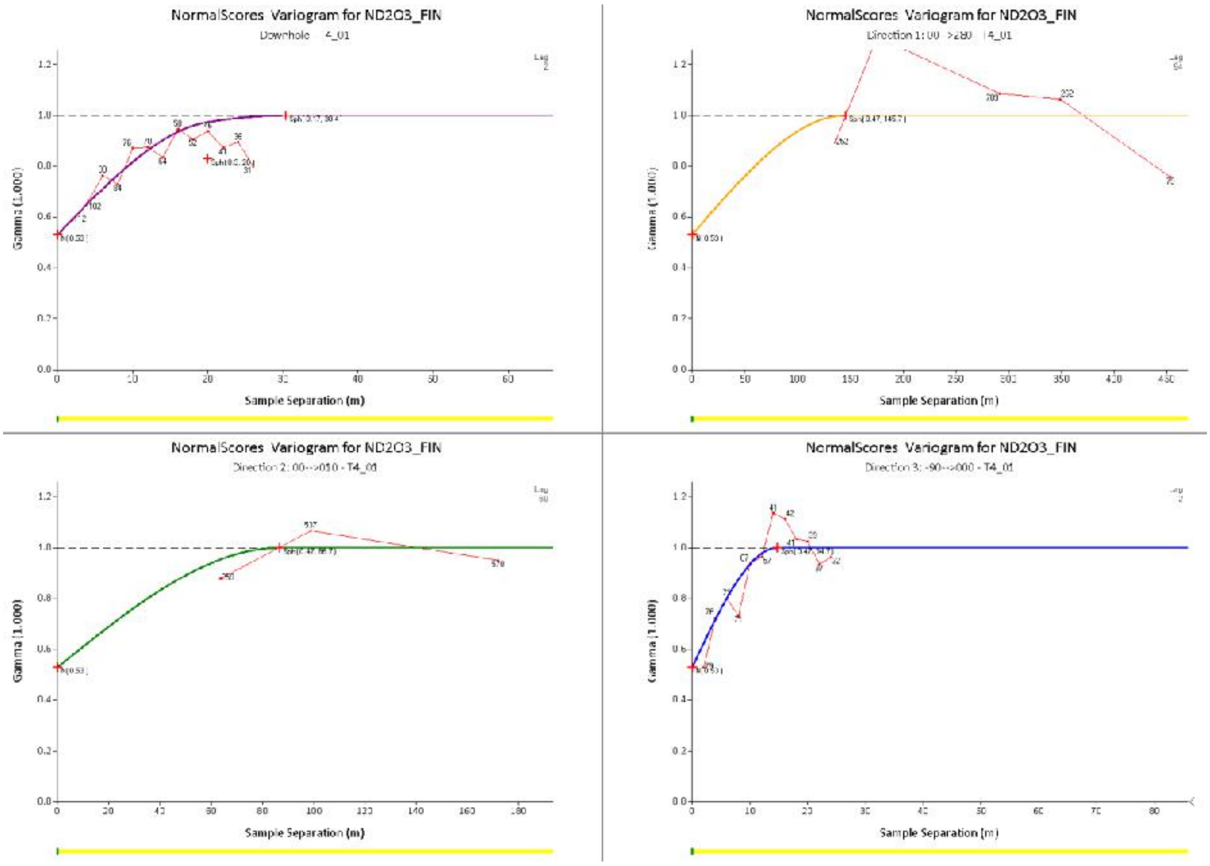

BackTransform Model for CE2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.264
Ranges in Direction
Sill 1 2 3
Spherical 0.241 221.6 178.3 9.6
Spherical 0.495 271.8 184.2 56.5

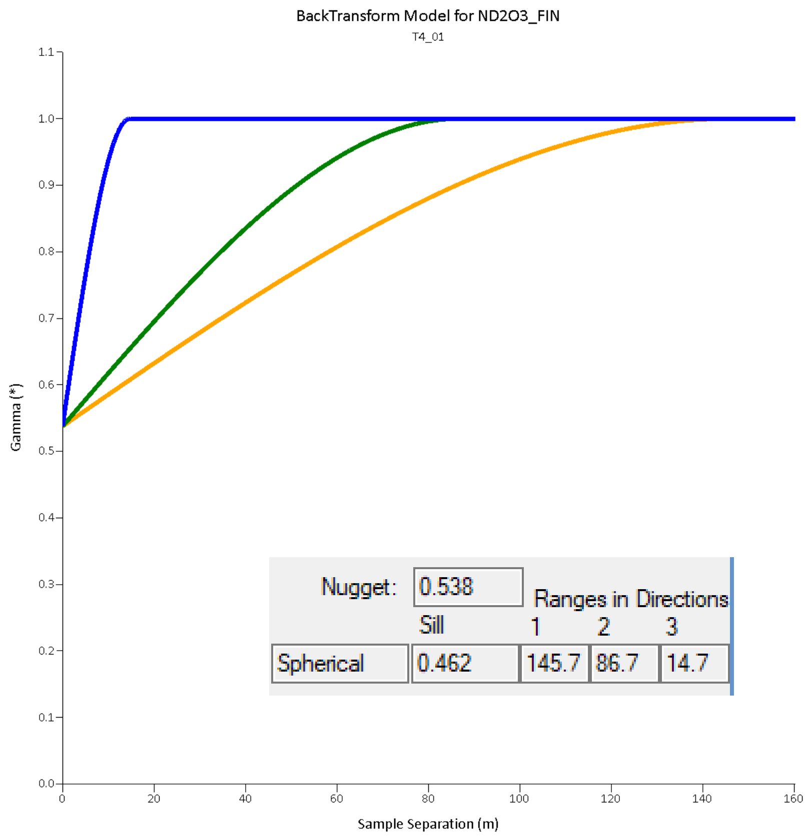
Dy_2O_3


NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Direction 2: -05-->209 - T4_01
Sample Separation (m)
Gamma (1.000)
NormalScores Variogram for DY2O3_FIN
Direction 3: 60-->290 - T4_01
Sample Separation (m)
Gamma (1.000)

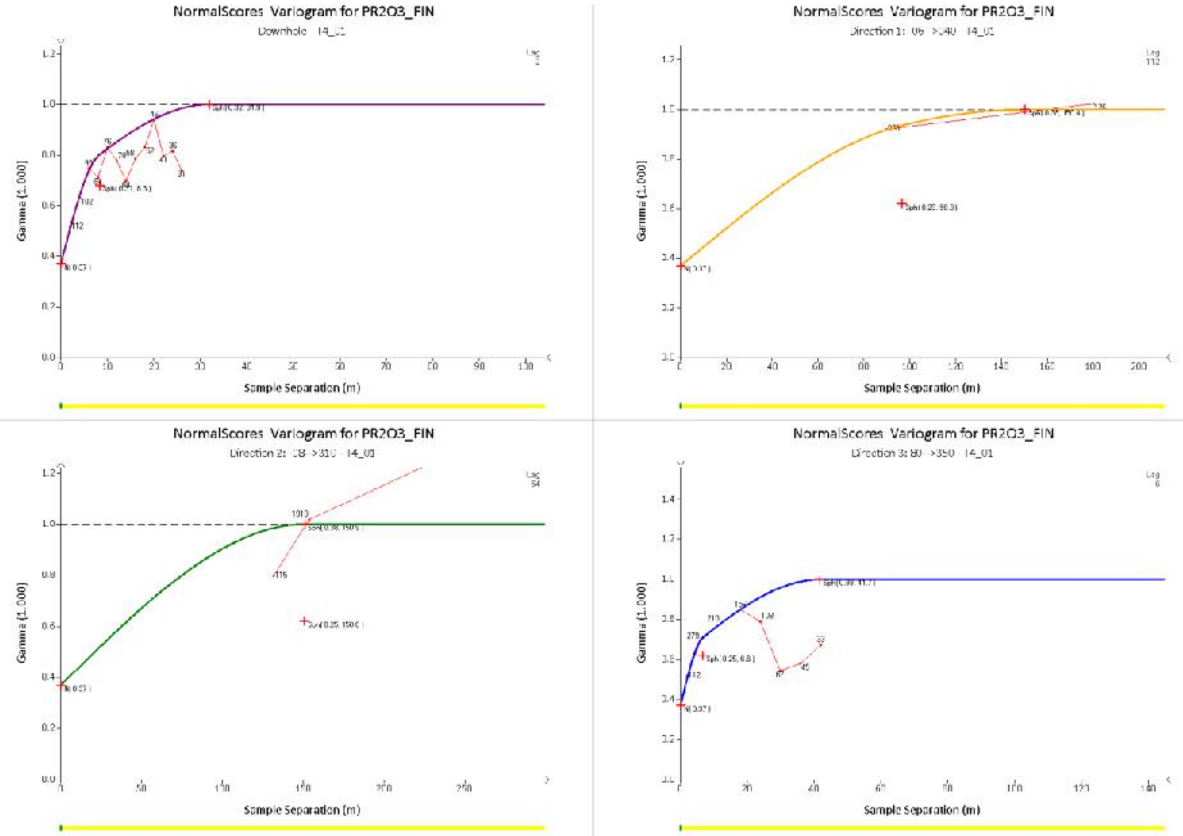

BackTransform Model for DY2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.184
Ranges in Directions
Sill 1 2 3
Spherical 0.816 187.4 105.1 48.9

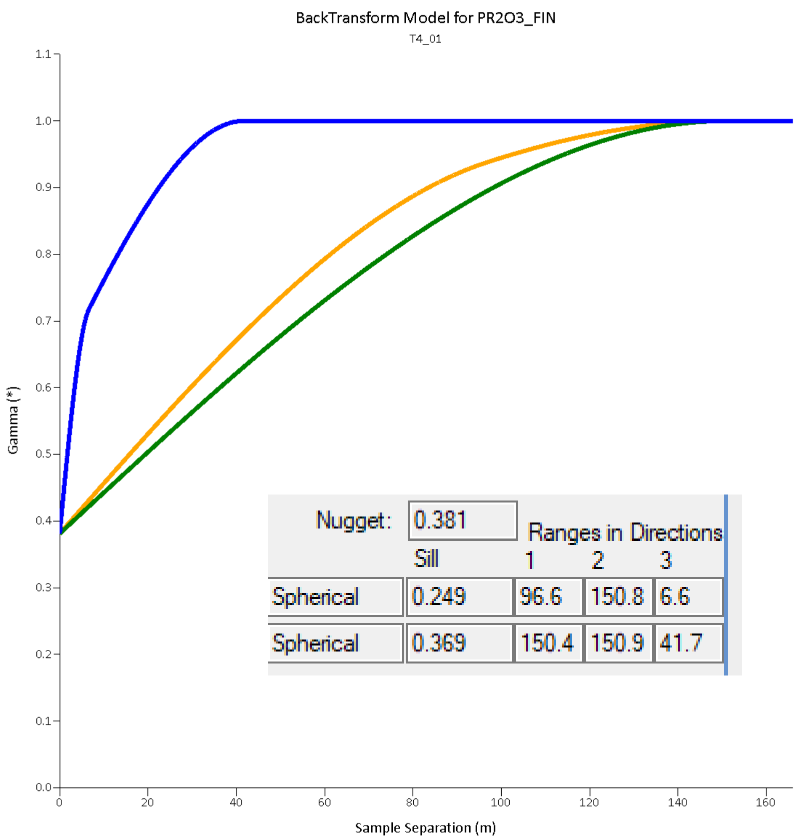
Nd_2O_3


NormalScores Variogram for ND2O3_FIN
Downhole - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for ND2O3_FIN
Direction 1: 00-->280 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for ND2O3_FIN
Direction 2: 00-->010 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for ND2O3_FIN
Direction 3: -90-->000 - T4_01
Gamma (1.000)
Sample Separation (m)

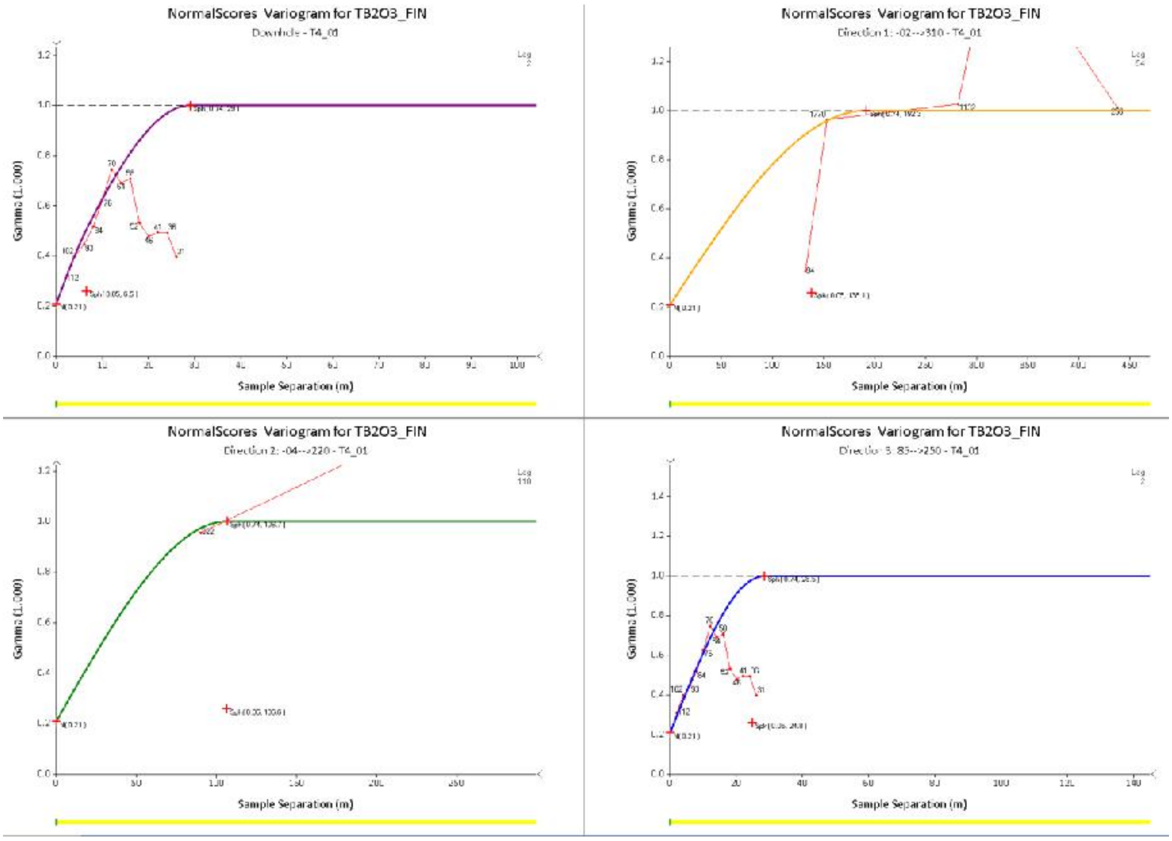

BackTransform Model for ND2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.538
Ranges in Directions
Sill
1 2 3
Spherical 0.462 145.7 86.7 14.7

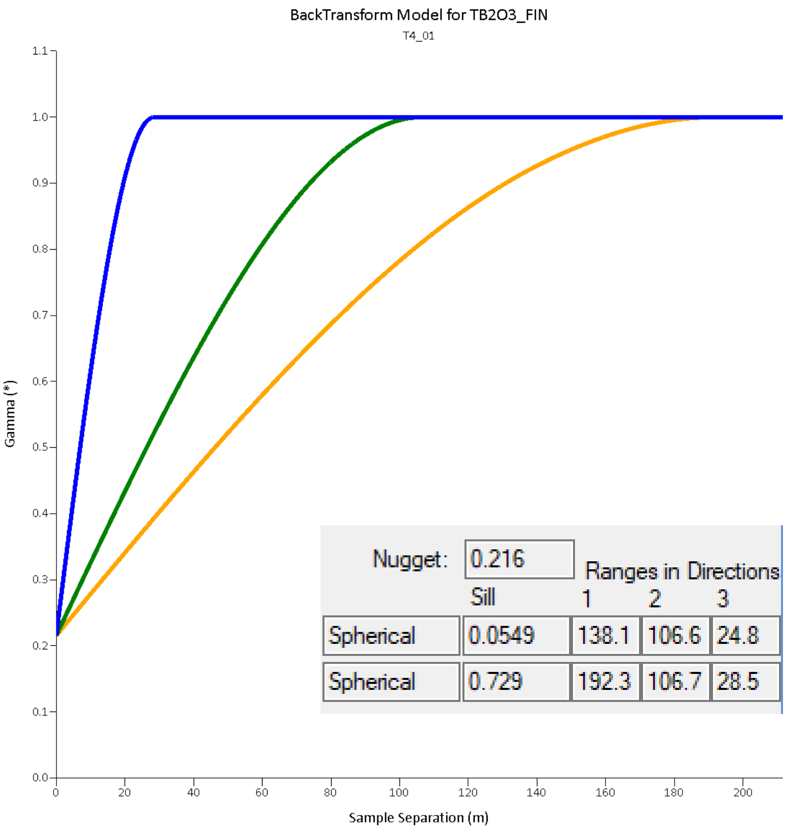
Pr_2O_3


NormalScores Variogram for PR2O3_FIN
Downhole T4_01
Direction 1: 06 ->240 T4_01
Direction 2: -08 ->310 T4_01
Direction 3: 80 ->350 T4_01
Gamma (1.000)
Sample Separation (m)

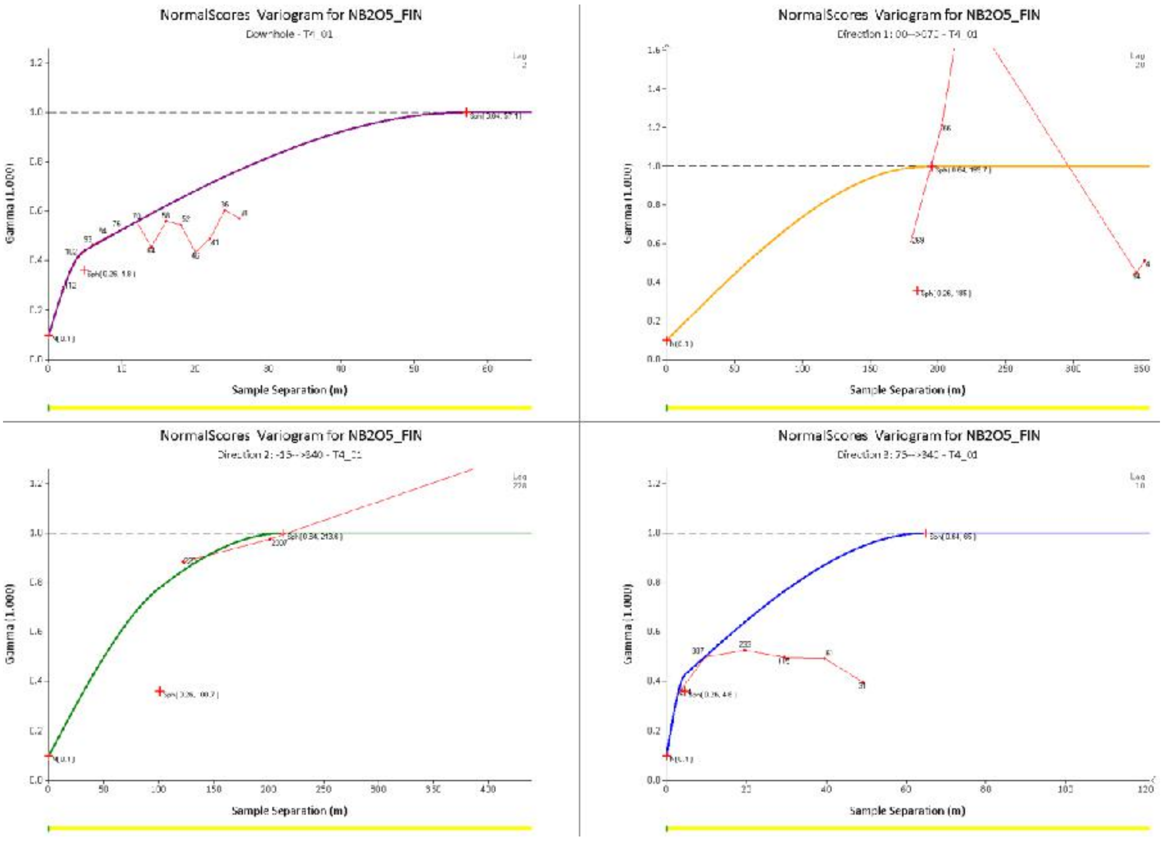

BackTransform Model for PR2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.381
Ranges in Directions
Sill 1 2 3
Spherical 0.249 96.6 150.8 6.6
Spherical 0.369 150.4 150.9 41.7

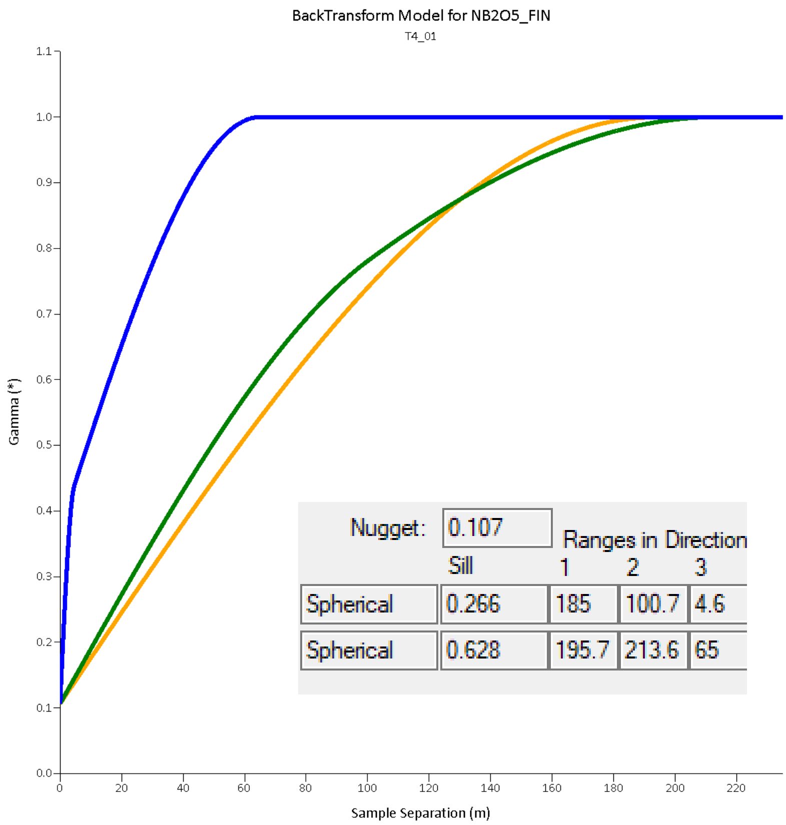
Tb_2O_3


NormalScores Variogram for TB2O3_FIN
Downhole - T4_01
NormalScores Variogram for TB2O3_FIN
Direction 1: -02-->310 - T4_01
NormalScores Variogram for TB2O3_FIN
Direction 2: -04-->220 - T4_01
NormalScores Variogram for TB2O3_FIN
Direction 3: 83-->250 - T4_01
Gamma (1.000)
Sample Separation (m)

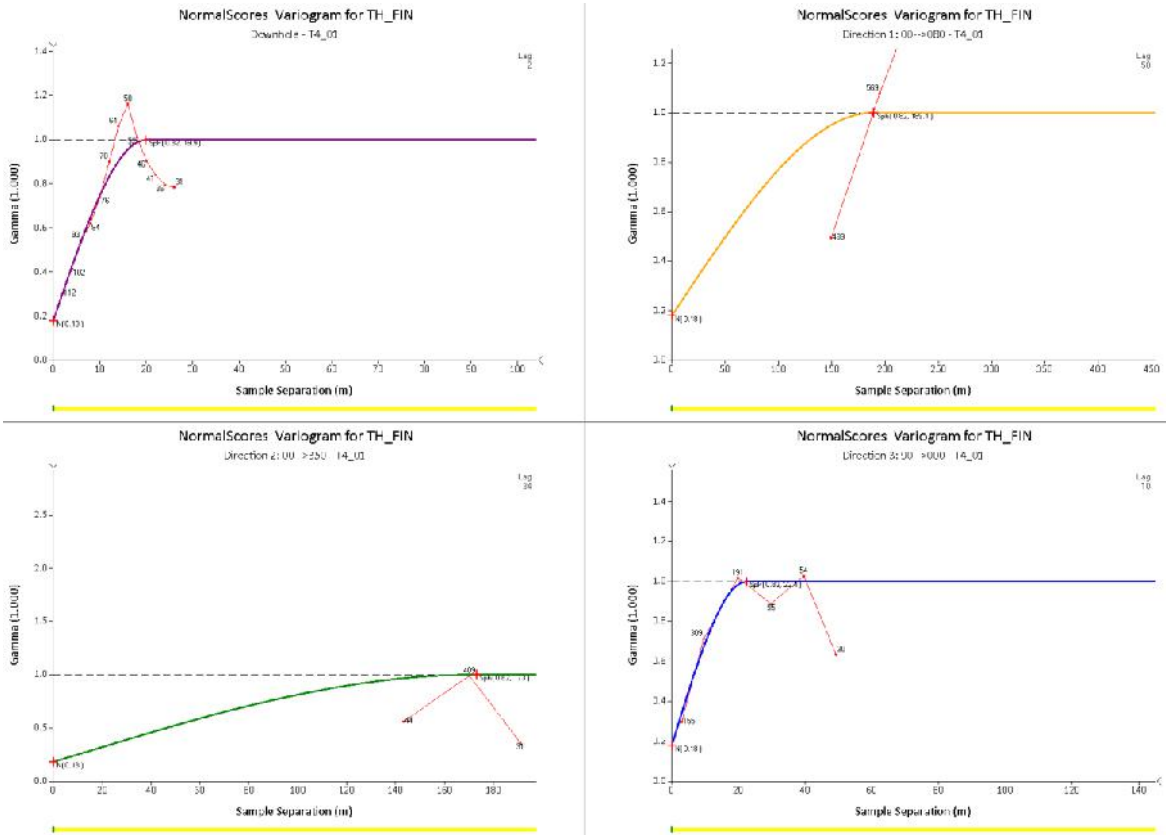

BackTransform Model for TB2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.216
Sill
Ranges in Directions
1 2 3
Spherical 0.0549 138.1 106.6 24.8
Spherical 0.729 192.3 106.7 28.5

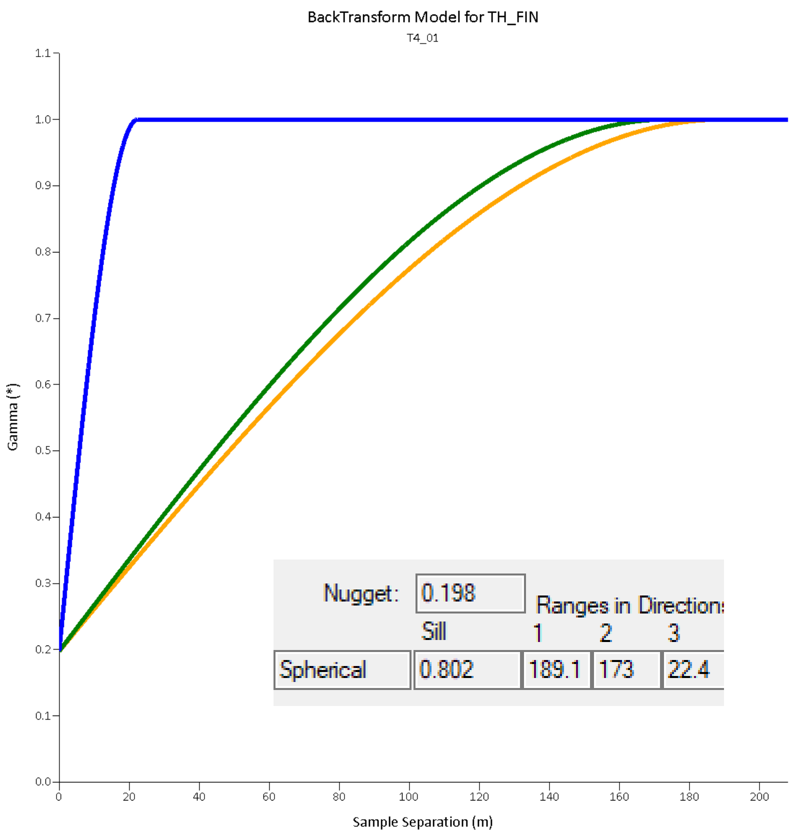
Nb_2O_5


NormalScores Variogram for NB2O5_FIN
Downhole - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 1: 00-->070 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 2: -15-->340 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Direction 3: 75-->340 - T4_01
Gamma (1.000)
Sample Separation (m)

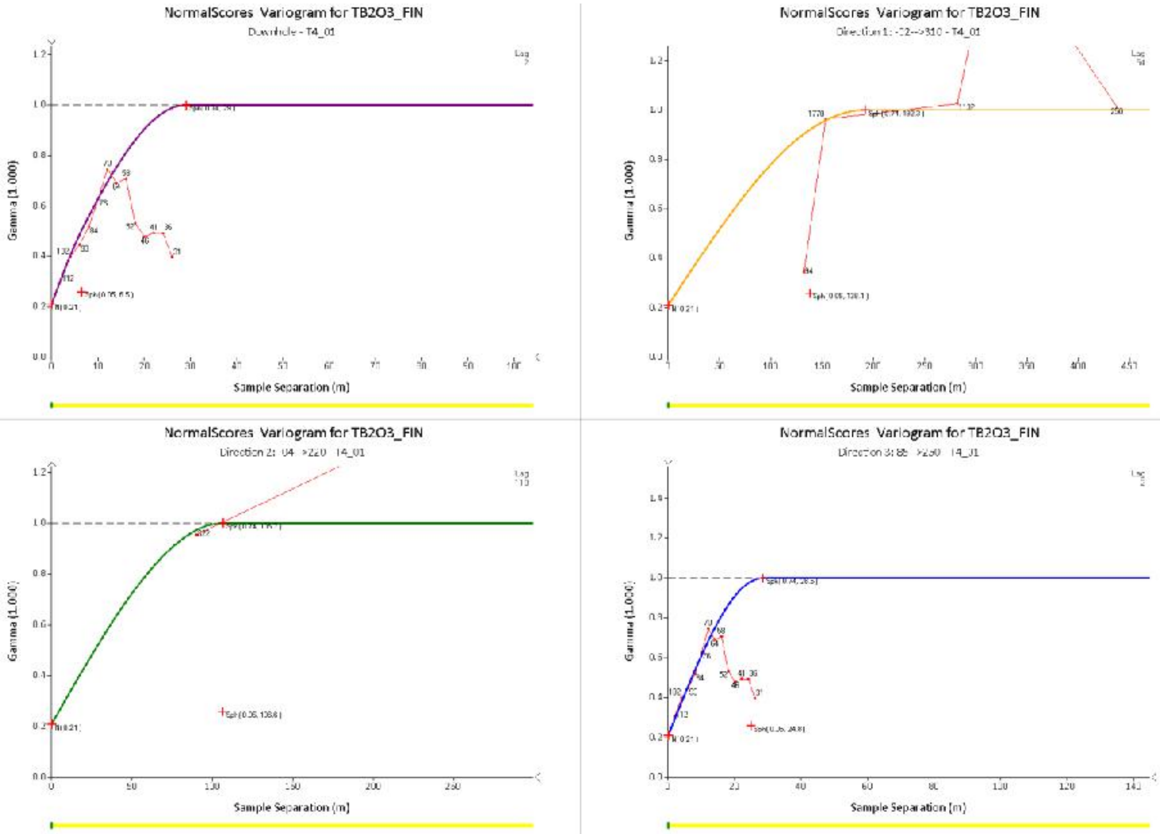

BackTransform Model for NB2O5_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.107
Ranges in Direction
Sill 1 2 3
Spherical 0.266 185 100.7 4.6
Spherical 0.628 195.7 213.6 65

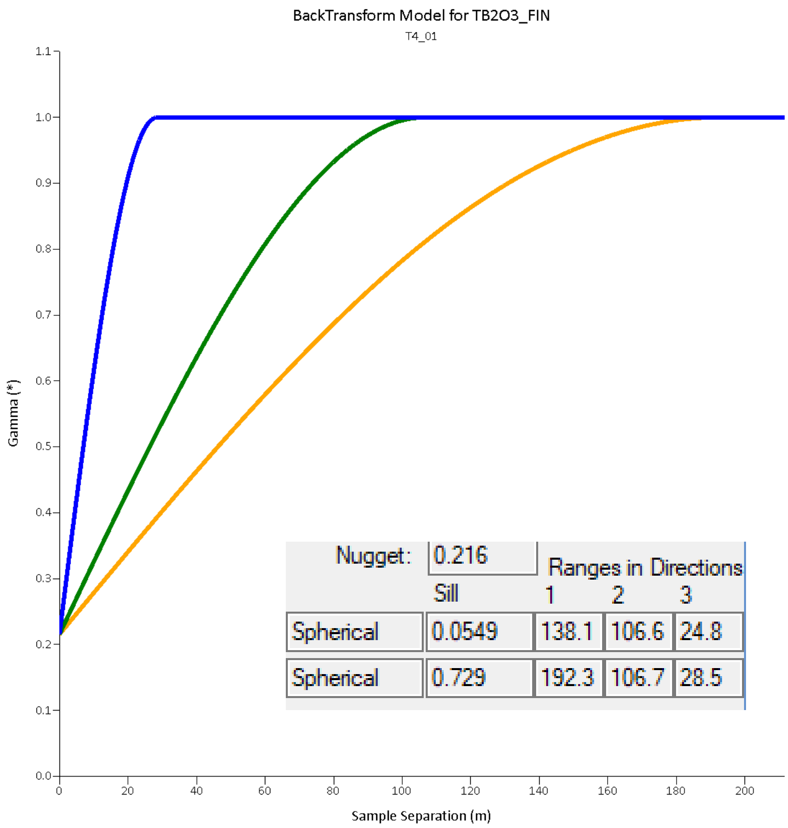
Th


NormalScores Variogram for TH_FIN
Downhole - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TH_FIN
Direction 1: 00-->080 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TH_FIN
Direction 2: 00-->350 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TH_FIN
Direction 3: 90-->000 - T4_01
Gamma (1.000)
Sample Separation (m)


BackTransform Model for TH_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.198
Ranges in Directions
Sill 1 2 3
Spherical 0.802 189.1 173 22.4

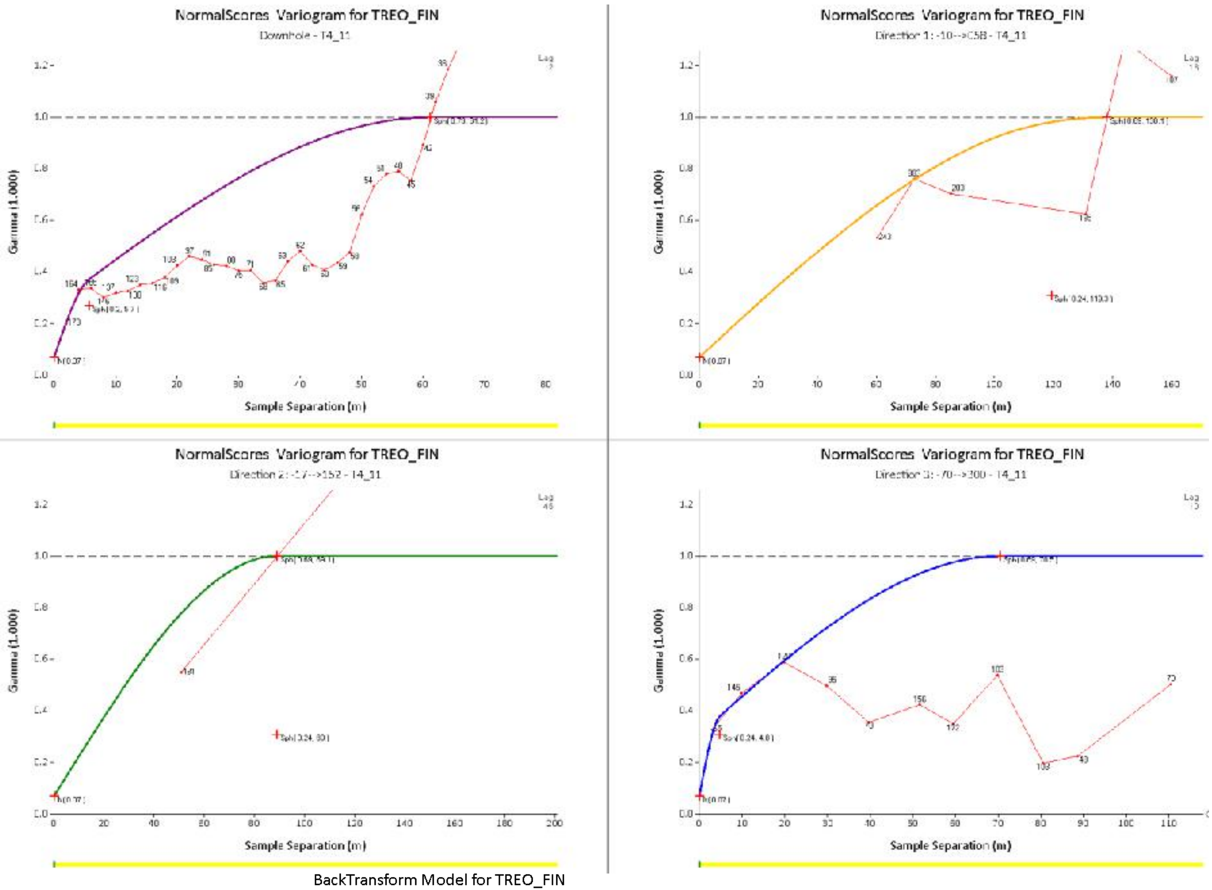
U

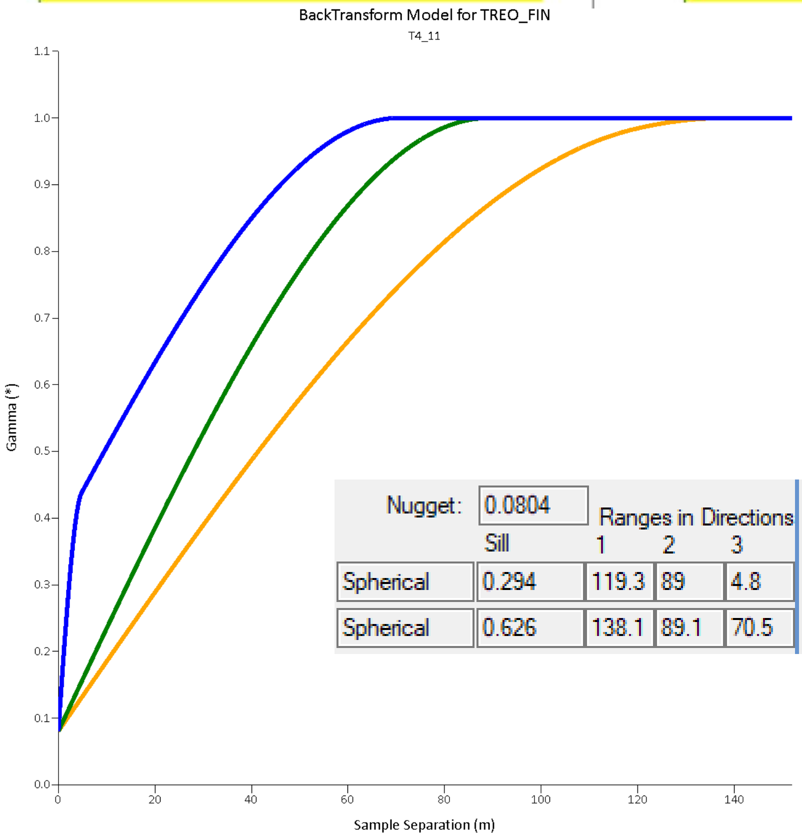

NormalScores Variogram for TB2O3_FIN
Downhole - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 1: -02-->310 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 2: 04 -->220 - T4_01
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for TB2O3_FIN
Direction 3: 85 -->250 - T4_01
Gamma (1.000)
Sample Separation (m)


BackTransform Model for TB2O3_FIN
T4_01
Gamma (*)
Sample Separation (m)
Nugget: 0.216
Ranges in Directions
Sill 1 2 3
Spherical 0.0549 138.1 106.6 24.8
Spherical 0.729 192.3 106.7 28.5

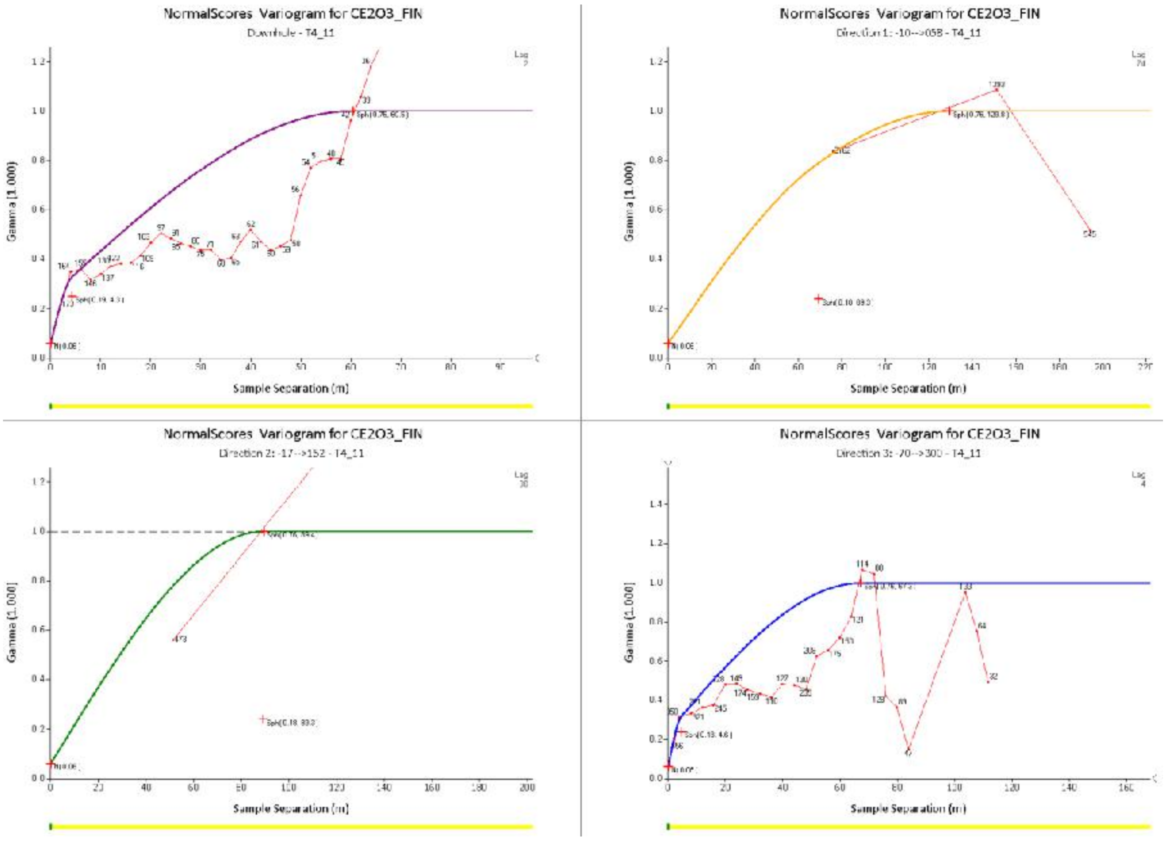
Target 1 Variogram models ≥1.0% TREO domain (ESTDOM T4_11)

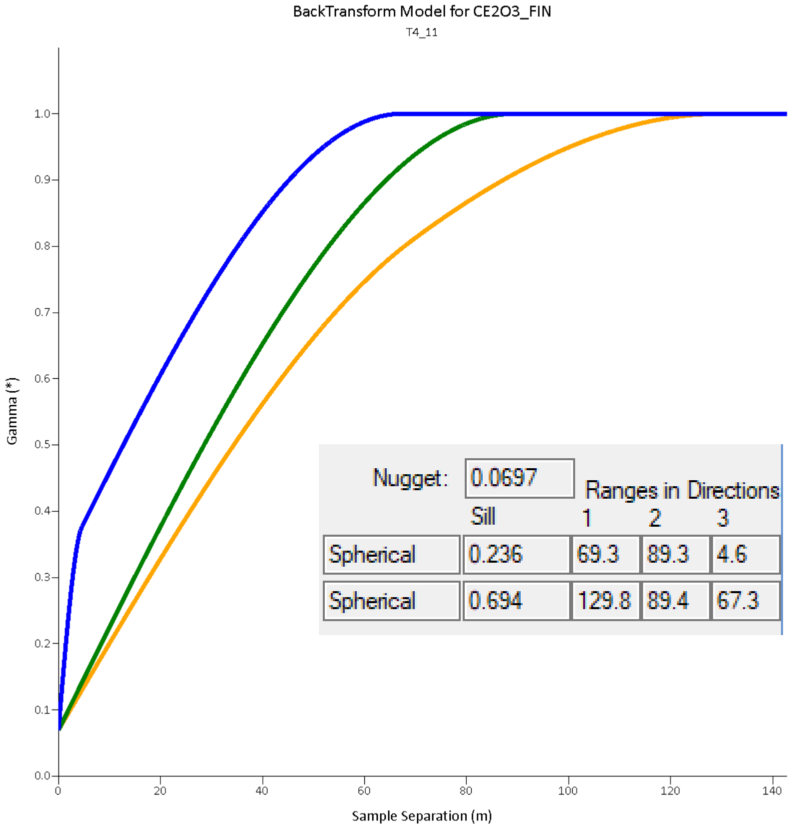
TREO





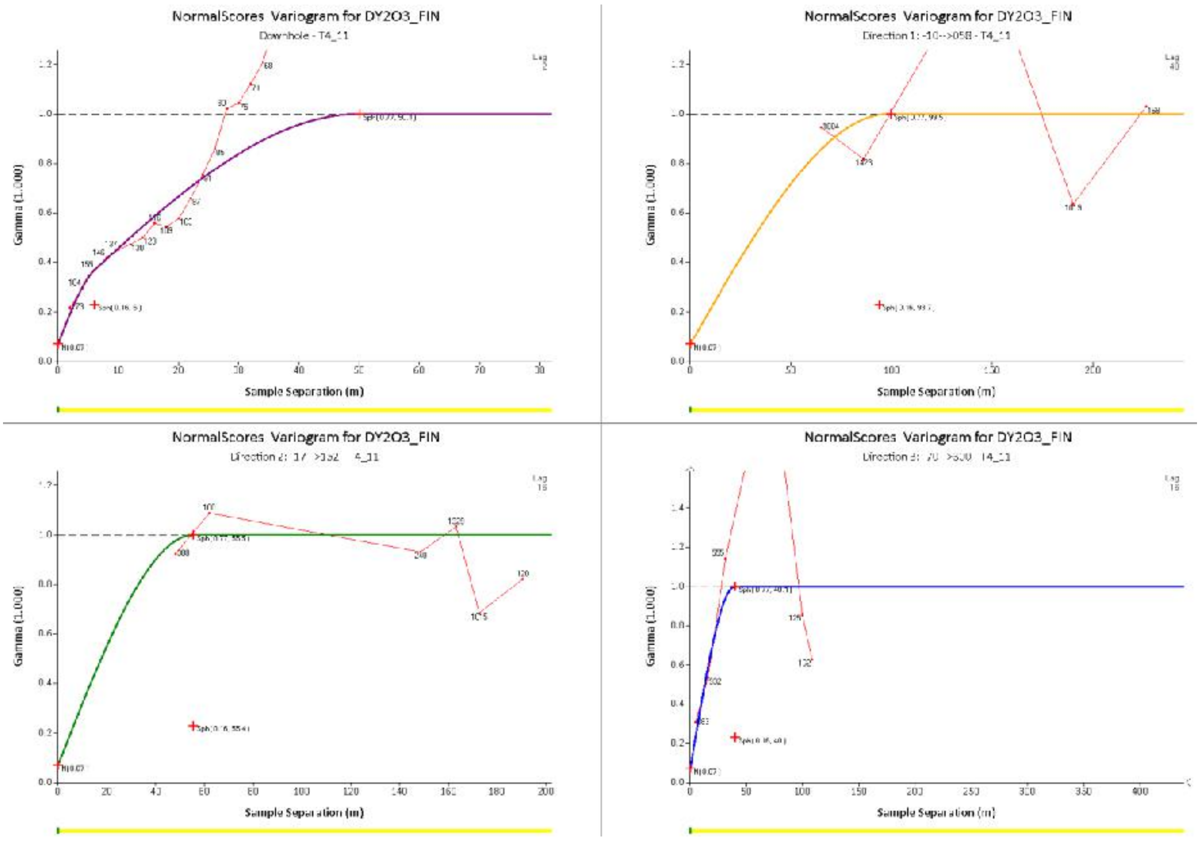
Ce_2O_3

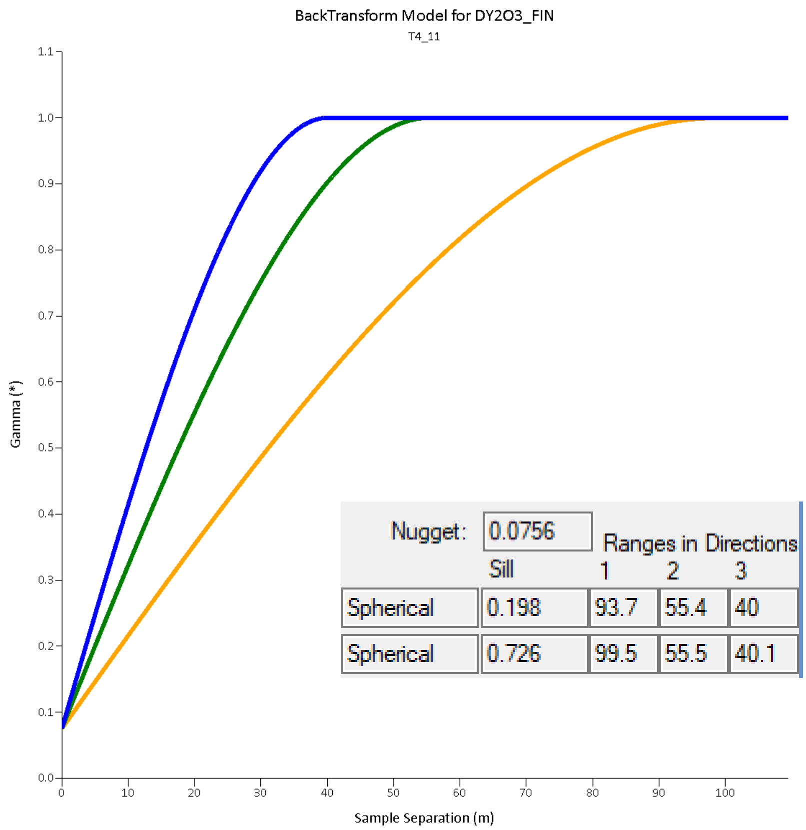




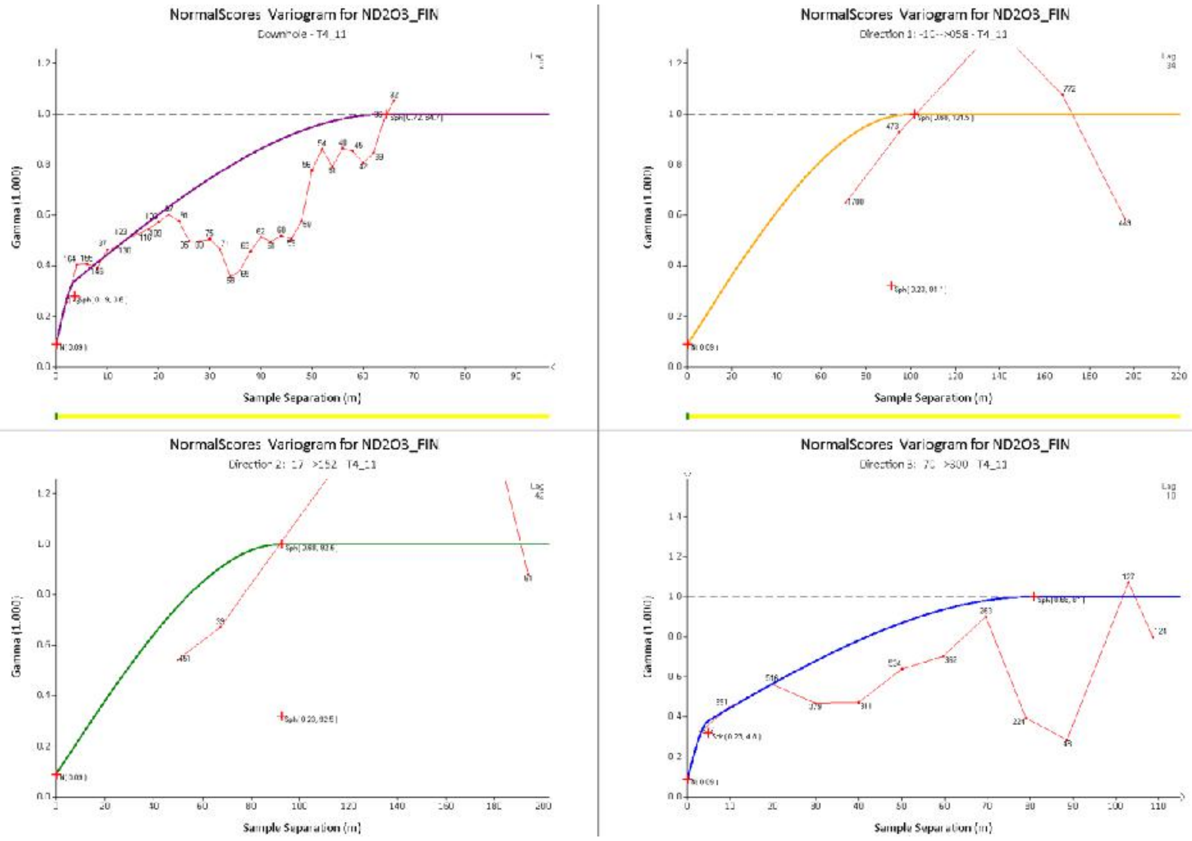
Nugget:	0.0697	Ranges in Directions		
	Sill	1	2	3
Spherical	0.236	69.3	89.3	4.6
Spherical	0.694	129.8	89.4	67.3

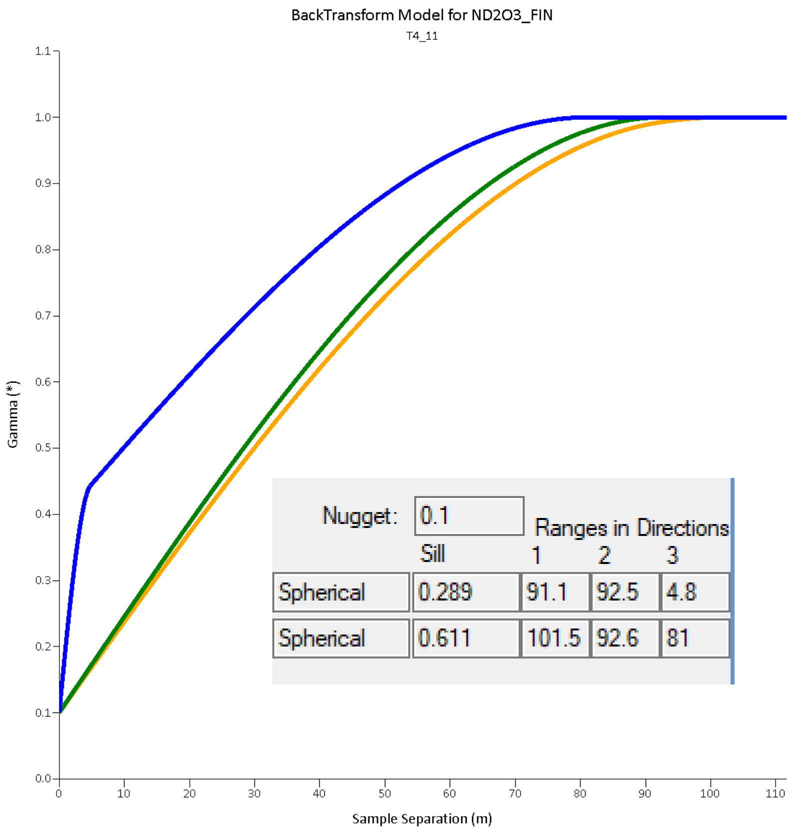
Dy_2O_3


NormalScores Variogram for DY2O3_FIN
Downhole - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Direction 1: -10-->058 - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for DY2O3_FIN
Gamma (1.000)
Sample Separation (m)

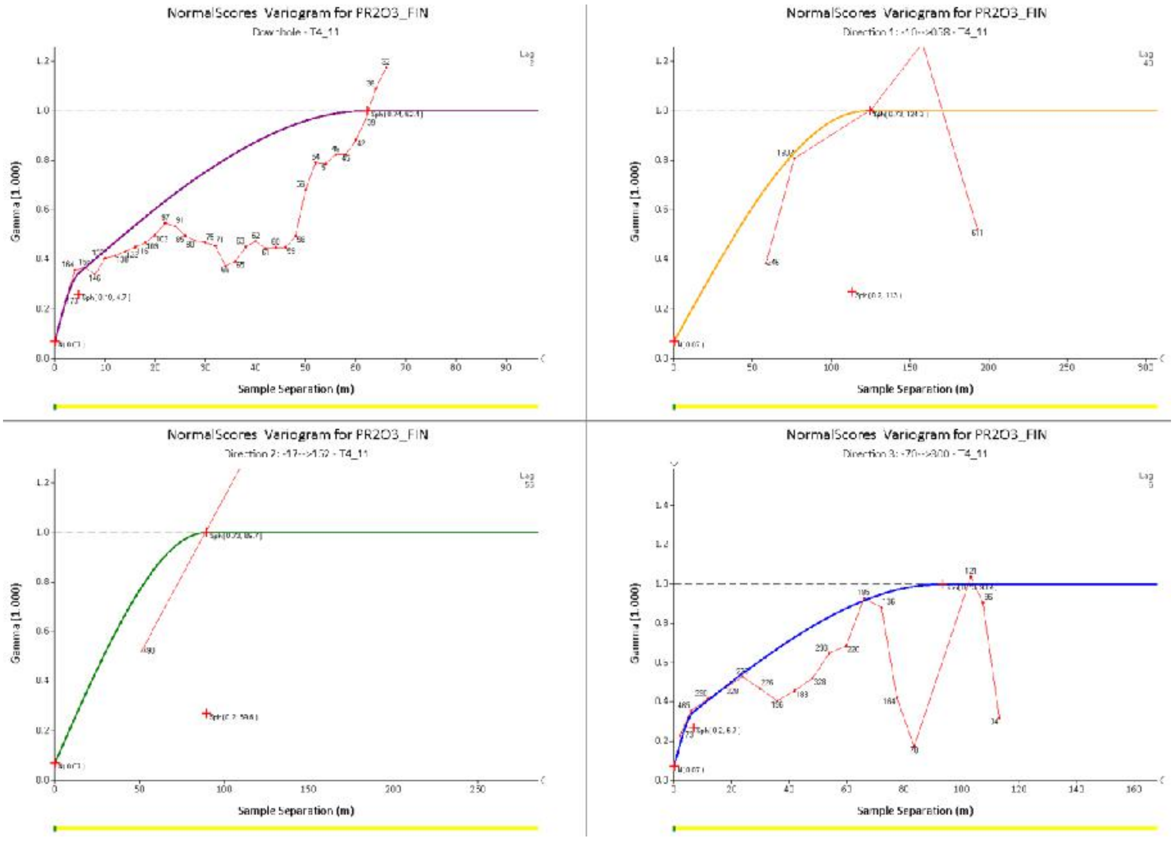

BackTransform Model for DY2O3_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.0756
Ranges in Directions
Sill 1 2 3
Spherical 0.198 93.7 55.4 40
Spherical 0.726 99.5 55.5 40.1

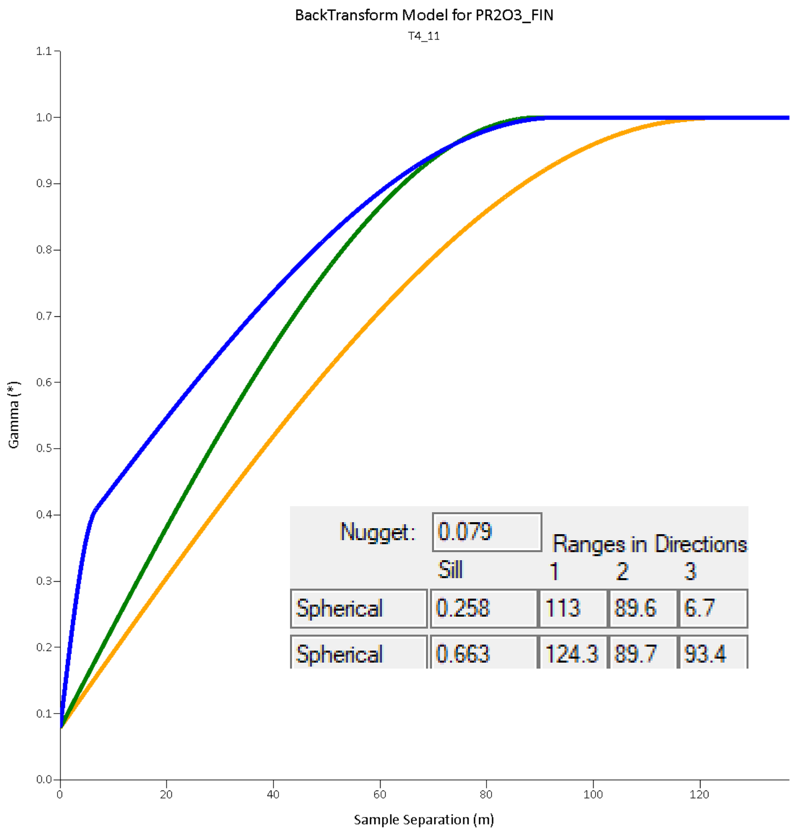
Nd_2O_3


NormalScores Variogram for ND2O3_FIN
Downhole - T4_11
Direction 1: -10-->058 - T4_11
Direction 2: 17 >152 T4_11
Direction 3: 73 >300 T4_11
Gamma (1.000)
Sample Separation (m)

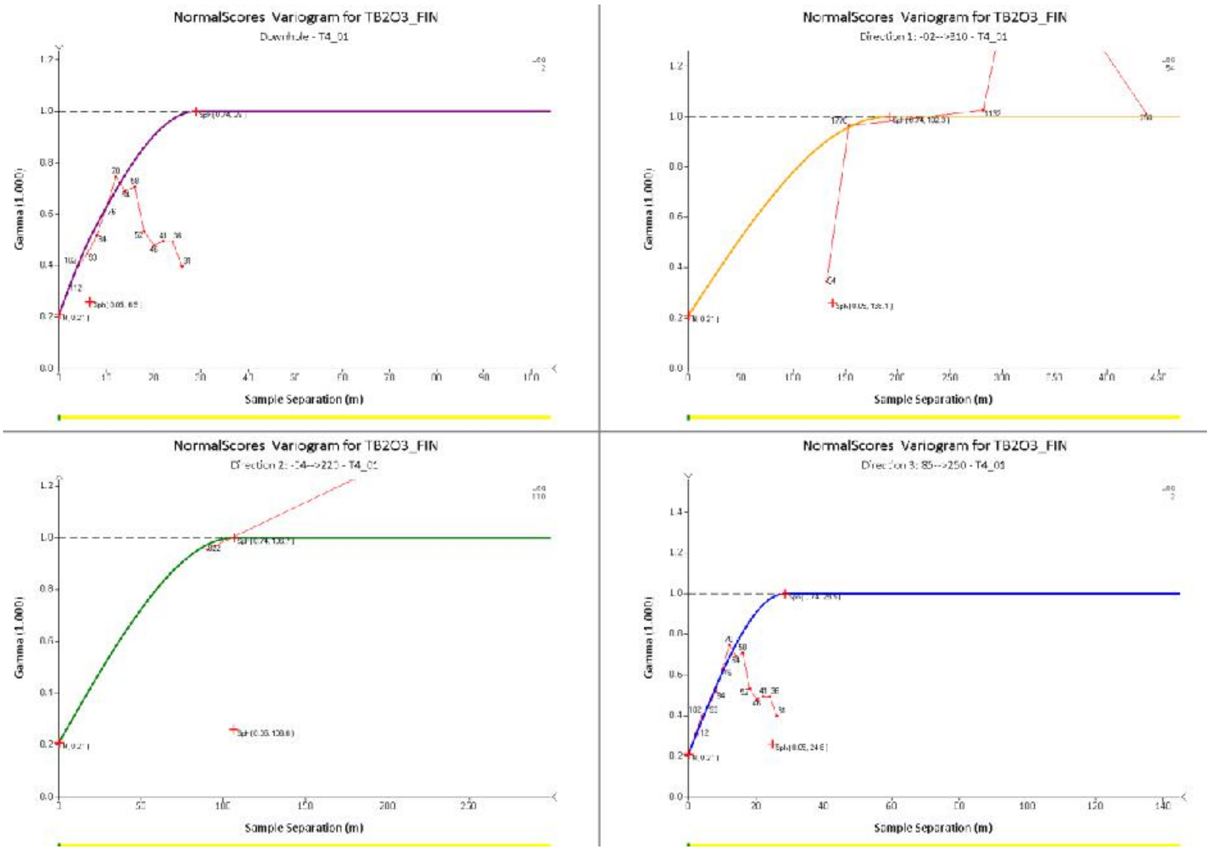

BackTransform Model for ND2O3_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.1
Ranges in Directions
Sill 1 2 3
Spherical 0.289 91.1 92.5 4.8
Spherical 0.611 101.5 92.6 81

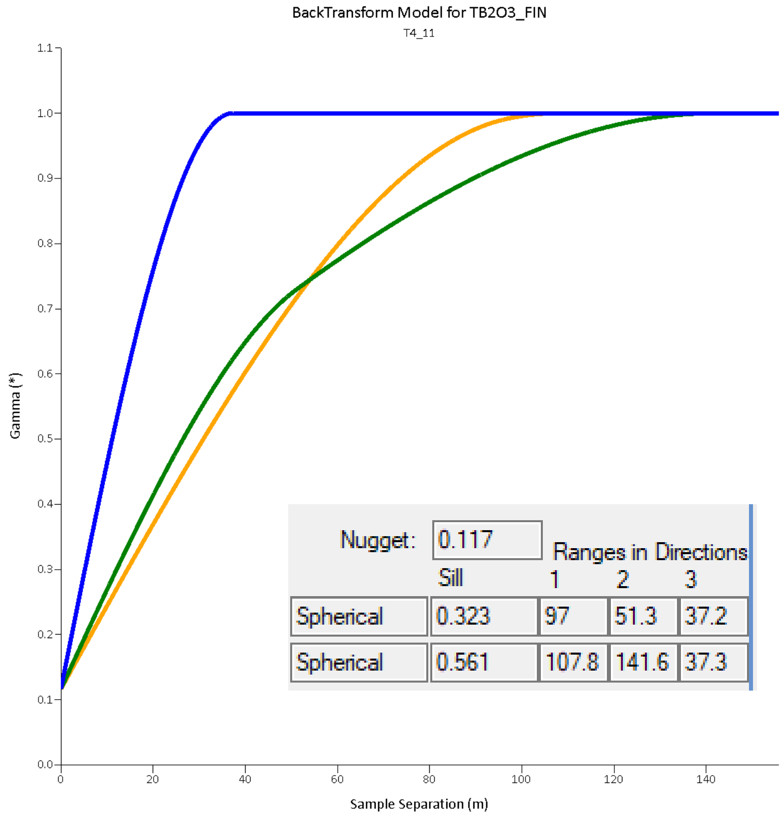
Pr_2O_3


NormalScores Variogram for PR2O3_FIN
Downhole - T4_11
NormalScores Variogram for PR2O3_FIN
Direction 1: -10-->038 - T4_11
NormalScores Variogram for PR2O3_FIN
Direction 2: -17-->132 - T4_11
NormalScores Variogram for PR2O3_FIN
Direction 3: -70-->300 - T4_11
Gamma (1.000)
Sample Separation (m)

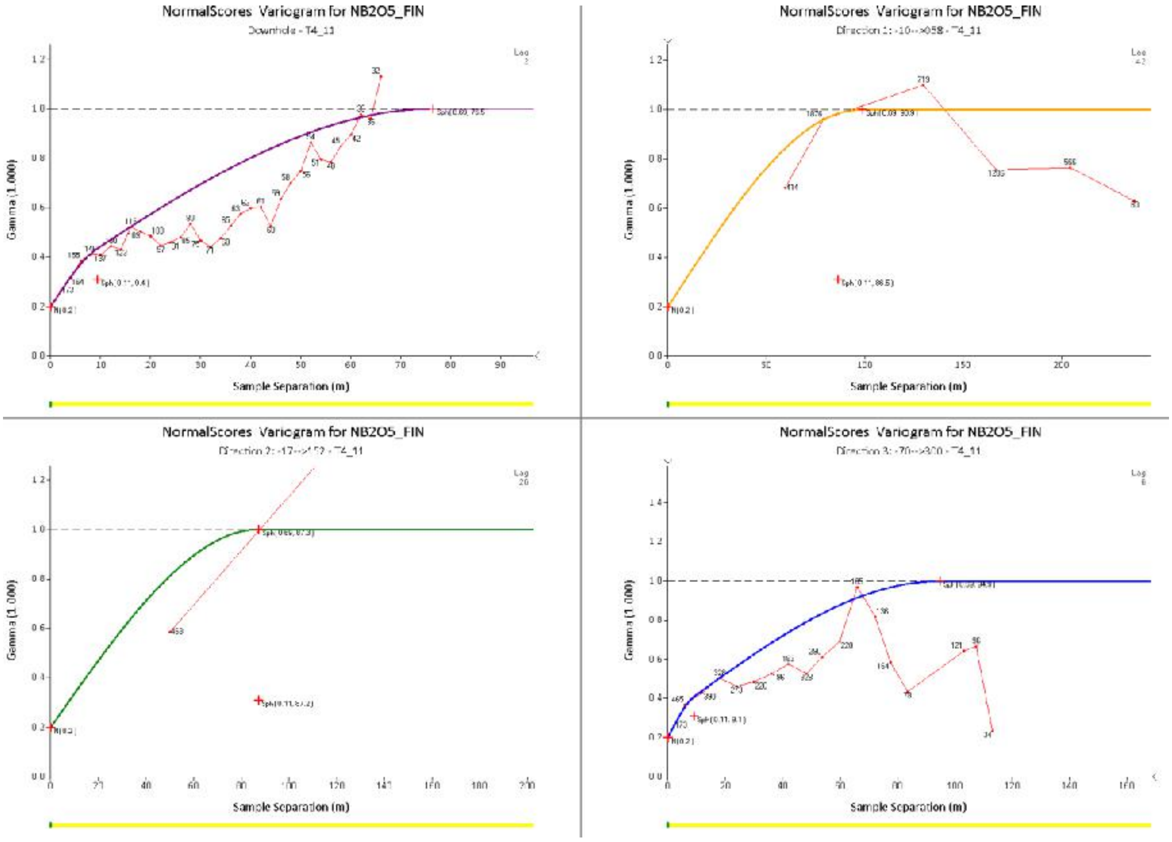

BackTransform Model for PR2O3_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.079
Ranges in Directions
Sill 1 2 3
Spherical 0.258 113 89.6 6.7
Spherical 0.663 124.3 89.7 93.4

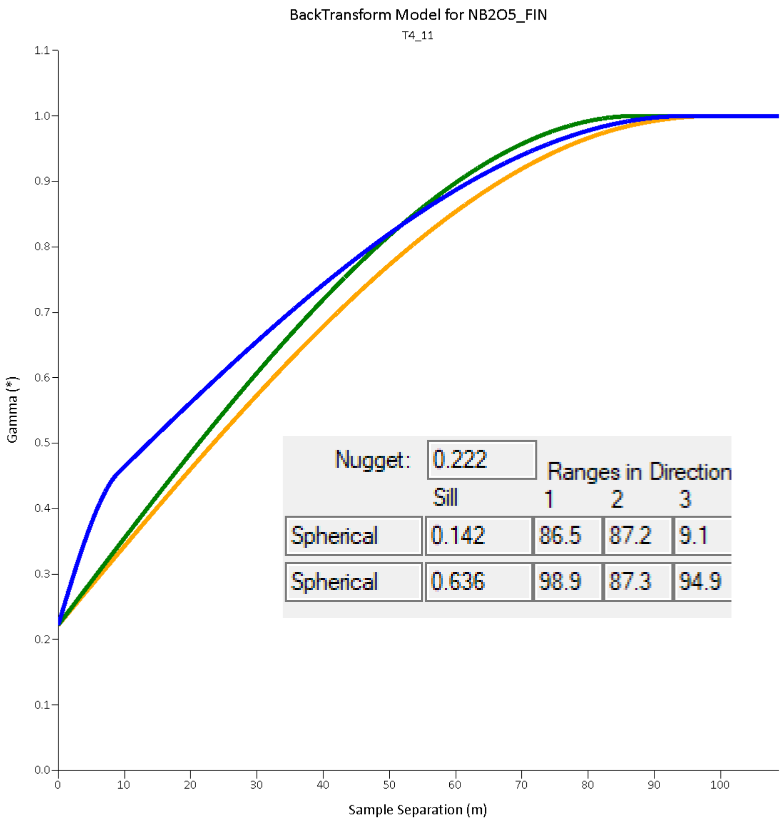
Tb_2O_3


NormalScores Variogram for TB2O3_FIN
Gamma (1.000)
Sample Separation (m)

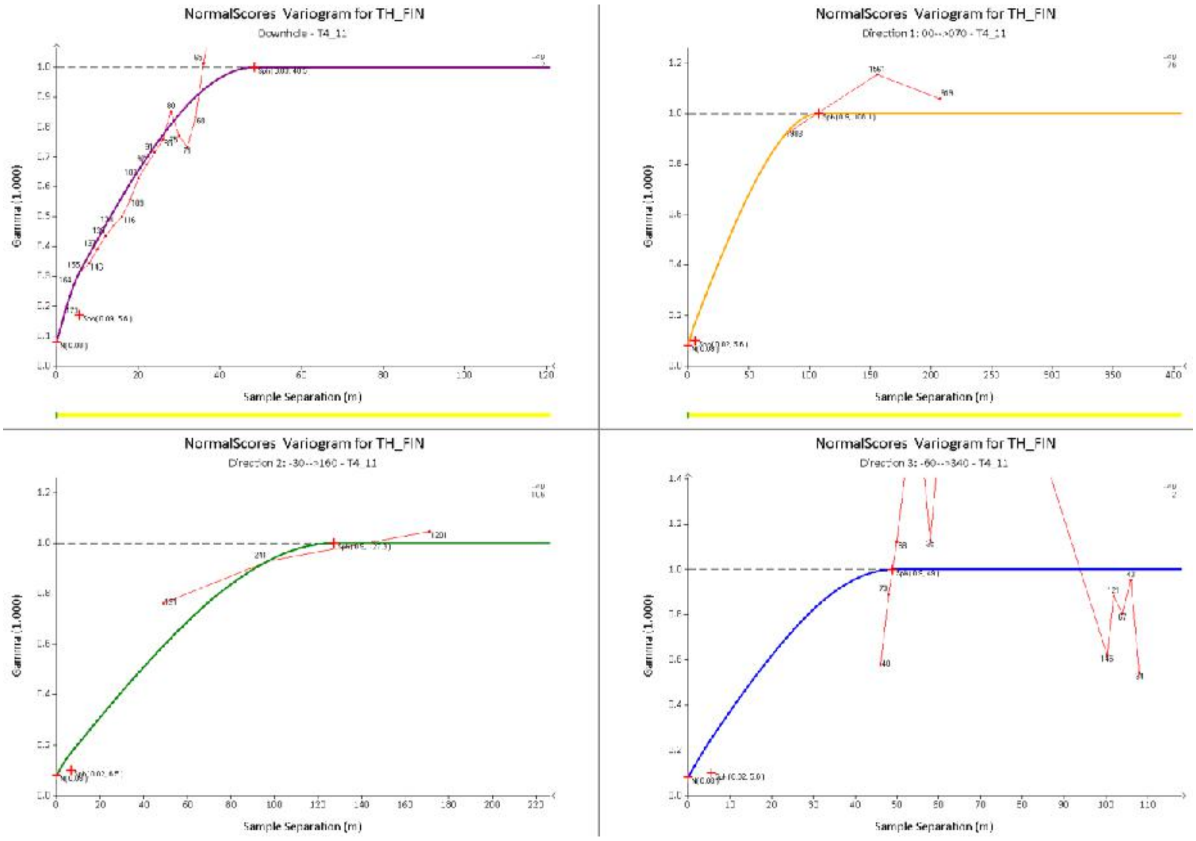

BackTransform Model for TB2O3_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.117
Ranges in Directions
Sill 1 2 3
Spherical 0.323 97 51.3 37.2
Spherical 0.561 107.8 141.6 37.3

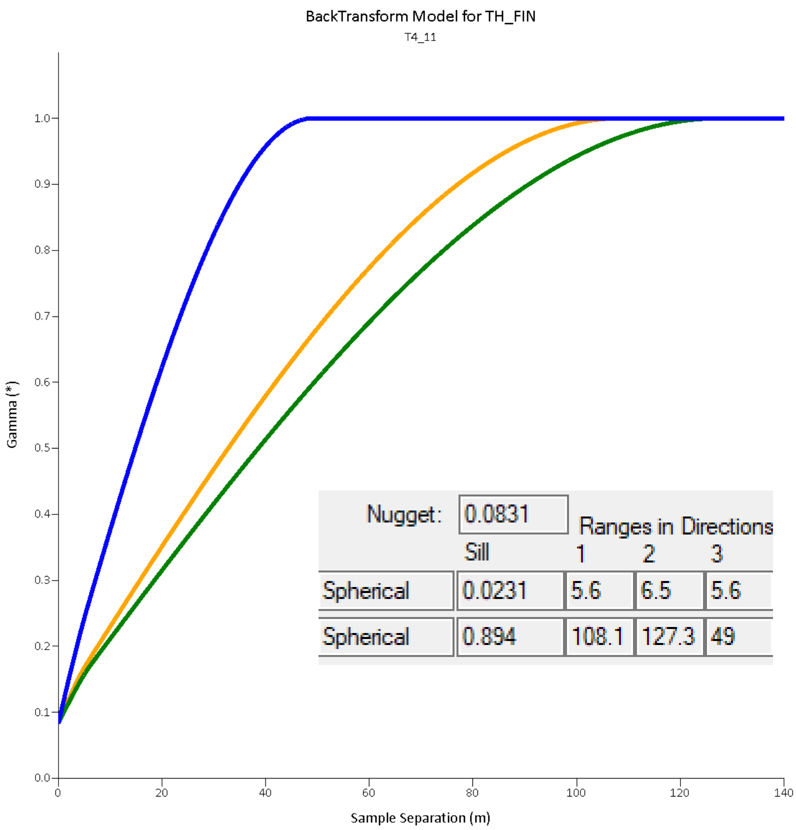
Nb_2O_5


NormalScores Variogram for NB2O5_FIN
Downhole - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for NB2O5_FIN
Gamma (1.000)
Sample Separation (m)

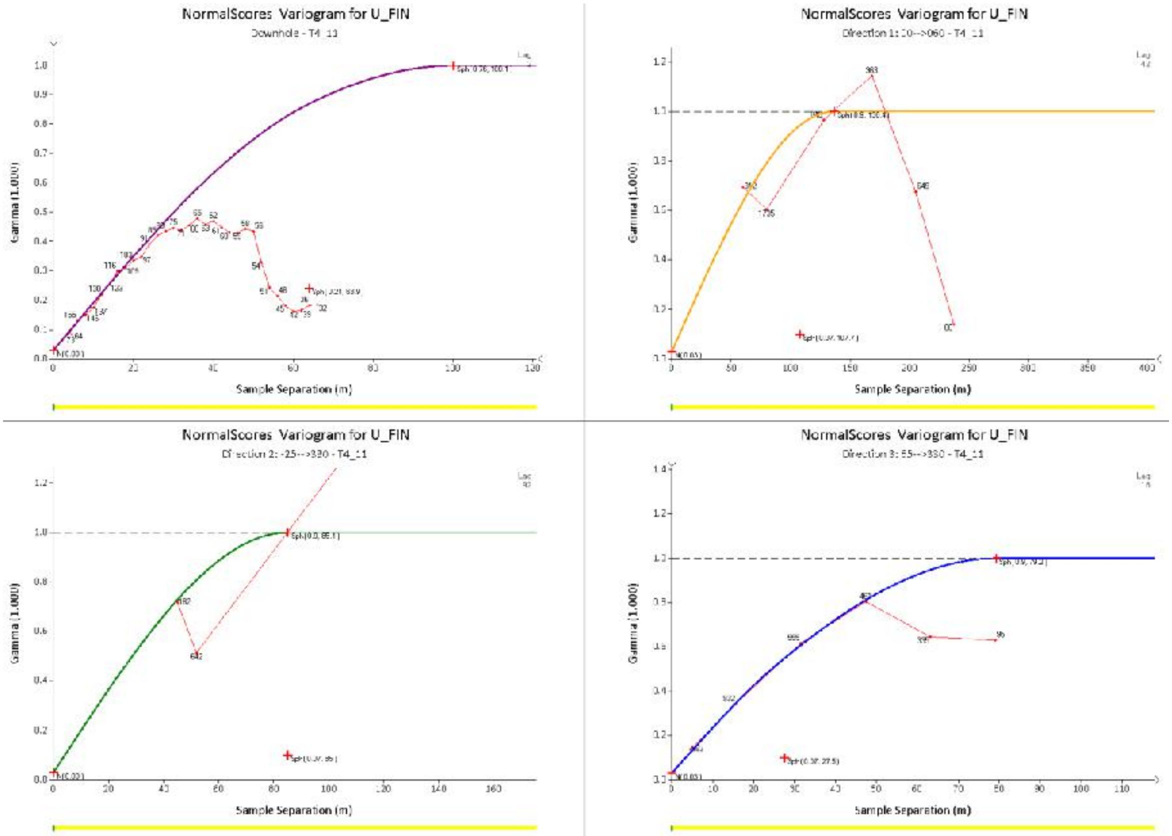

BackTransform Model for NB2O5_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.222
Ranges in Direction
Sill 1 2 3
Spherical 0.142 86.5 87.2 9.1
Spherical 0.636 98.9 87.3 94.9

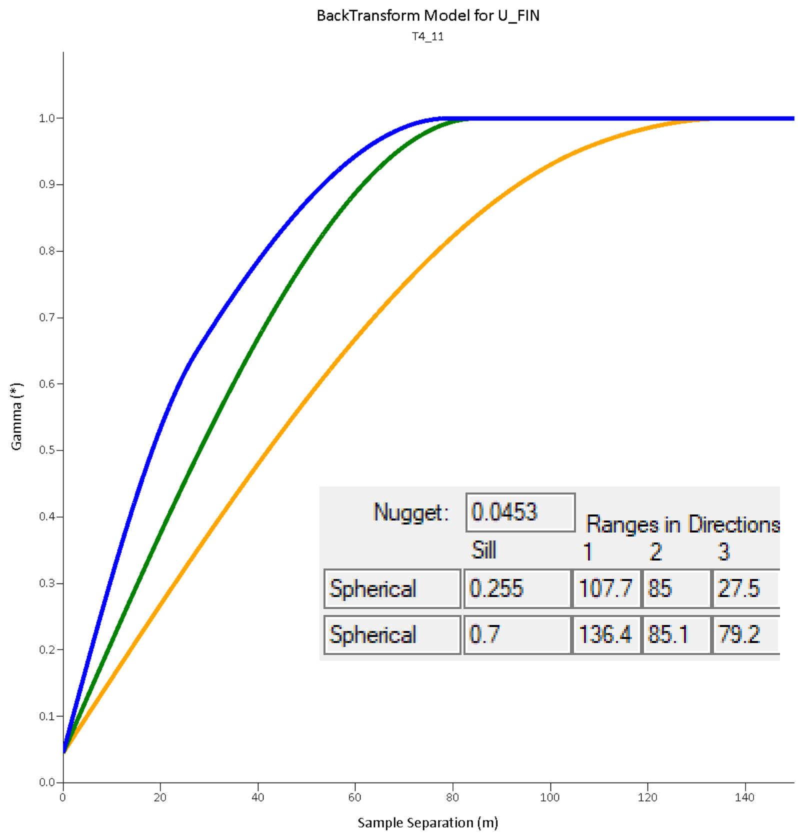
Th


NormalScores Variogram for TH_FIN
Downhole - T4_11
NormalScores Variogram for TH_FIN
Direction 1: 00-->070 - T4_11
NormalScores Variogram for TH_FIN
Direction 2: -30-->160 - T4_11
NormalScores Variogram for TH_FIN
Direction 3: -60-->340 - T4_11
Gamma (1.000)
Sample Separation (m)


BackTransform Model for TH_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.0831
Ranges in Directions
Sill 1 2 3
Spherical 0.0231 5.6 6.5 5.6
Spherical 0.894 108.1 127.3 49

U


NormalScores Variogram for U_FIN
Downhole - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for U_FIN
Direction 1: 10-->060 - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for U_FIN
Direction 2: -25-->330 - T4_11
Gamma (1.000)
Sample Separation (m)
NormalScores Variogram for U_FIN
Direction 3: 65-->330 - T4_11
Gamma (1.000)
Sample Separation (m)


BackTransform Model for U_FIN
T4_11
Gamma (*)
Sample Separation (m)
Nugget: 0.0453
Ranges in Directions
Sill 1 2 3
Spherical 0.255 107.7 85 27.5
Spherical 0.7 136.4 85.1 79.2

Appendix F

Validation of the Mineral Resource Estimate

Appendix 5 – Whole of domain average grade comparison

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	7,104	7,463	7,290	2.6%	-2.3%
	T1_02	14	7,488	7,369	7,494	0.1%	1.7%
	T1_03	42	5,917	5,973	6,059	2.4%	1.4%
	T1_04	36	11,549	10,315	11,096	-3.9%	7.6%
	T1_11	584	23,174	22,600	23,543	1.6%	4.2%
	T1_13	40	29,418	26,507	29,183	-0.8%	10.1%
	T1_14	68	14,934	14,690	14,726	-1.4%	0.2%
TREO	T1_15	20	15,560	15,528	14,066	-9.6%	-9.4%
ppm	T1_16	20	13,468	14,296	13,368	-0.7%	-6.5%
	T1_17	14	11,292	11,324	11,639	3.1%	2.8%
	T1_18	8	12,453	12,248	12,293	-1.3%	0.4%
	T4_01	122	7,384	7,207	7,768	5.2%	7.8%
	T4_11	182	21,270	20,166	20,243	-4.8%	0.4%
	T6_01	108	7,560	7,724	7,720	2.1%	-0.1%
	T6_11	115	21,026	19,827	21,951	4.4%	10.7%
	T6_12	25	24,397	23,948	23,342	-4.3%	-2.5%

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	13	13	14	0.9%	1.0%
	T1_02	14	29	28	29	0.3%	1.3%
	T1_03	42	14	14	14	4.1%	4.2%
	T1_04	36	11	11	11	4.3%	-0.5%
	T1_11	584	16	17	16	-2.6%	-8.2%
	T1_13	40	22	21	20	-7.5%	-1.4%
	T1_14	68	18	18	18	-0.3%	-0.8%
Tb_2O_3	T1_15	20	30	32	33	8.8%	1.7%
ppm	T1_16	20	26	25	25	-1.1%	1.1%
	T1_17	14	33	32	34	4.2%	5.7%
	T1_18	8	27	29	26	-4.4%	-10.4%
	T4_01	122	25	25	25	0.1%	0.4%
	T4_11	182	27	27	26	-6.3%	-6.4%
	T6_01	108	18	17	16	-8.3%	-2.7%
	T6_11	115	23	23	22	-4.2%	-3.4%
	T6_12	25	16	16	17	3.5%	3.3%

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	3,054	3,156	3,127	2.4%	-0.9%
	T1_02	14	2,746	2,730	2,757	0.4%	1.0%
	T1_03	42	2,492	2,605	2,563	2.8%	-1.6%
	T1_04	36	5,440	5,288	5,180	-4.8%	-2.0%
	T1_11	584	10,606	10,607	10,922	3.0%	3.0%
	T1_13	40	13,699	12,527	13,620	-0.6%	8.7%
	T1_14	68	6,703	7,011	6,749	0.7%	-3.7%
Ce_2O_3	T1_15	20	6,711	5,917	5,907	-12.0%	-0.2%
ppm	T1_16	20	5,778	5,841	5,700	-1.3%	-2.4%
	T1_17	14	4,460	4,444	4,603	3.2%	3.6%
	T1_18	8	5,212	5,378	5,206	-0.1%	-3.2%
	T4_01	122	2,822	2,781	2,979	5.6%	7.1%
	T4_11	182	9,125	8,599	8,632	-5.4%	0.4%
	T6_01	108	2,959	3,003	3,273	10.6%	9.0%
	T6_11	115	9,711	9,181	10,291	6.0%	12.1%
	T6_12	25	11,666	11,700	11,366	-2.6%	-2.8%

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	1,142	1,190	1,197	4.9%	0.6%
	T1_02	14	3,124	3,134	3,033	-2.9%	-3.2%
	T1_03	42	1,307	1,275	1,303	-0.4%	2.1%
	T1_04	36	673	663	645	-4.1%	-2.6%
	T1_11	584	581	665	620	6.8%	-6.8%
	T1_13	40	591	556	551	-6.8%	-0.9%
	T1_14	68	1,239	1,222	1,074	-13.3%	-12.1%
Nb_2O_5	T1_15	20	3,728	4,150	4,417	18.5%	6.4%
ppm	T1_16	20	1,137	1,062	1,165	2.4%	9.7%
	T1_17	14	2,033	2,017	2,132	4.9%	5.7%
	T1_18	8	1,096	1,114	1,093	-0.3%	-1.9%
	T4_01	122	1,255	1,255	1,203	-4.2%	-4.2%
	T4_11	182	688	692	715	3.9%	3.3%
	T6_01	108	828	901	966	16.6%	7.3%
	T6_11	115	779	822	729	-6.4%	-11.3%
	T6_12	25	278	277	290	4.6%	4.9%

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	71	71	71	1.1%	1.2%
	T1_02	14	156	155	159	2.0%	2.7%
	T1_03	42	75	75	80	6.5%	6.6%
	T1_04	36	57	60	61	6.0%	0.6%
	T1_11	584	83	88	82	-0.4%	-6.9%
	T1_13	40	118	109	108	-8.4%	-0.5%
	T1_14	68	91	92	91	-0.6%	-1.3%
Dy_2O_3	T1_15	20	137	134	134	-2.4%	-0.2%
ppm	T1_16	20	137	134	134	-2.4%	-0.2%
	T1_17	14	182	179	185	1.4%	2.9%
	T1_18	8	149	161	140	-6.1%	-13.1%
	T4_01	122	139	139	139	-0.1%	-0.5%
	T4_11	182	149	150	140	-6.2%	-6.8%
	T6_01	108	92	87	86	-6.3%	-1.0%
	T6_11	115	120	118	114	-5.0%	-3.7%
	T6_12	25	78	78	81	3.6%	3.5%

Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	149	148	153	2.2%	3.3%
	T1_02	14	253	249	254	0.5%	2.2%
	T1_03	42	183	186	188	3.0%	1.3%
	T1_04	36	257	263	269	4.7%	2.2%
	T1_11	584	202	212	201	-0.7%	-5.3%
	T1_13	40	719	714	681	-5.3%	-4.7%
	T1_14	68	170	183	168	-1.1%	-7.9%
Th ppm	T1_15	20	290	284	272	-6.1%	-4.2%
	T1_16	20	369	394	366	-0.9%	-7.1%
	T1_17	14	446	438	472	5.9%	7.8%
	T1_18	8	722	747	718	-0.5%	-3.8%
	T4_01	122	413	410	398	-3.7%	-3.2%
	T4_11	182	355	362	333	-6.3%	-8.1%
	T6_01	108	297	265	245	-17.5%	-7.6%
	T6_11	115	441	428	413	-6.4%	-3.4%
	T6_12	25	227	228	240	5.8%	5.5%

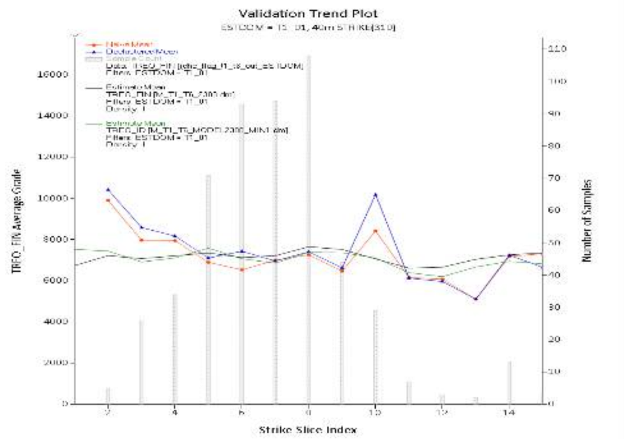
Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	1,007	1,021	1,021	1.4%	0.0%
	T1_02	14	1,292	1,280	1,280	-0.9%	0.0%
	T1_03	42	944	963	944	0.0%	-2.0%
	T1_04	36	1,948	1,920	1,776	-8.8%	-7.5%
	T1_11	584	2,209	2,255	2,248	1.8%	-0.3%
	T1_13	40	3,503	3,288	3,417	-2.5%	3.9%
	T1_14	68	1,811	1,815	1,803	-0.5%	-0.6%
Nd_2O_3	T1_15	20	2,117	2,065	2,131	0.6%	3.2%
ppm	T1_16	20	2,119	2,124	2,104	-0.7%	-0.9%
	T1_17	14	1,744	1,731	1,778	2.0%	2.7%
	T1_18	8	1,706	1,742	1,707	0.1%	-2.0%
	T4_01	122	939	934	962	2.4%	2.9%
	T4_11	182	1,986	1,942	1,873	-5.7%	-3.6%
	T6_01	108	1,214	1,195	1,236	1.8%	3.4%
	T6_11	115	2,972	2,822	3,070	3.3%	8.8%
	T6_12	25	2,605	2,615	2,480	-4.8%	-5.2%

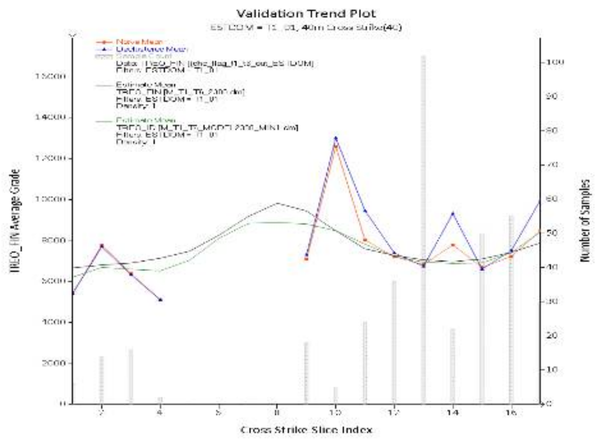
Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	15	16	17	11.4%	9.6%
	T1_02	14	92	91	87	-5.0%	-3.7%
	T1_03	42	25	24	25	-0.2%	2.0%
	T1_04	36	14	15	15	4.4%	0.7%
	T1_11	584	19	21	21	9.0%	-1.4%
	T1_13	40	29	27	27	-6.2%	1.9%
	T1_14	68	16	17	15	-3.2%	-12.3%
U ppm	T1_15	20	29	29	27	-6.4%	-7.6%
	T1_16	20	32	30	30	-5.5%	0.7%
	T1_17	14	36	36	44	21.6%	21.8%
	T1_18	8	22	25	20	-11.4%	-20.7%
	T4_01	122	12	12	11	-4.7%	-4.5%
	T4_11	182	10	10	12	22.1%	18.9%
	T6_01	108	20	22	24	22.3%	10.4%
	T6_11	115	26	28	25	-2.8%	-9.7%
	T6_12	25	13	13	14	7.8%	7.8%

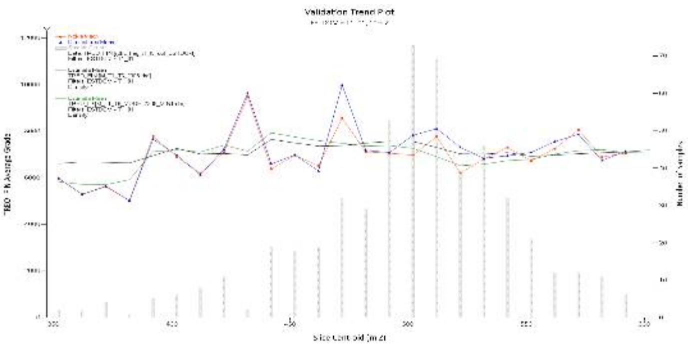
Variable	ESTDOM	Nos Samples	Composite Average		Model Av	Percent difference	
			Naïve Mean	Decl Mean		Naïve-model	Decl_Model
	T1_01	532	308	316	315	2.3%	-0.1%
	T1_02	14	330	328	336	1.6%	2.5%
	T1_03	42	274	281	283	3.2%	0.5%
	T1_04	36	623	609	582	-6.7%	-4.5%
	T1_11	584	862	874	874	1.4%	0.0%
	T1_13	40	1,219	1,154	1,206	-1.1%	4.5%
	T1_14	68	616	629	610	-1.0%	-3.0%
Pr_2O_3	T1_15	20	656	613	624	-4.9%	2.0%
ppm	T1_16	20	627	633	625	-0.3%	-1.2%
	T1_17	14	492	489	507	3.1%	3.7%
	T1_18	8	504	513	505	0.1%	-1.7%
	T4_01	122	279	276	290	3.7%	4.8%
	T4_11	182	739	738	697	-5.8%	-5.6%
	T6_01	108	346	347	366	5.5%	5.4%
	T6_11	115	963	911	999	3.7%	9.7%
	T6_12	25	971	974	926	-4.7%	-5.0%

Model validation trend plots target 1 - T1_01

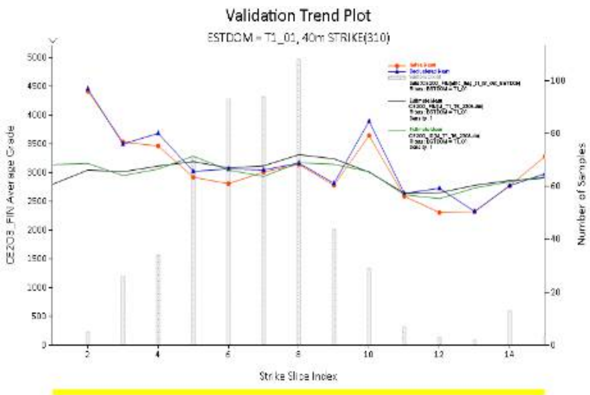
TREO

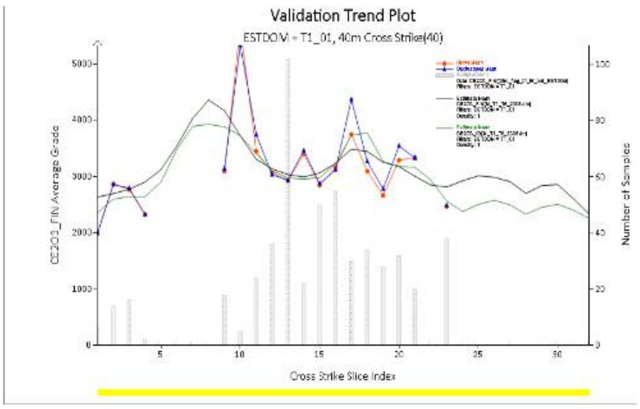


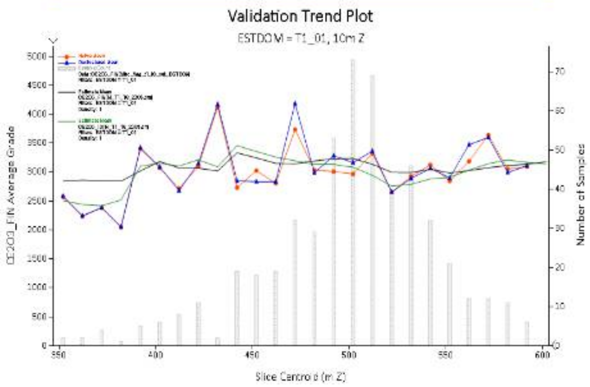




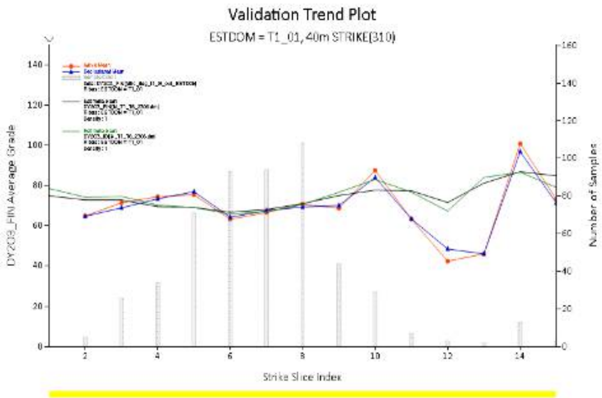
Ce_2O_3

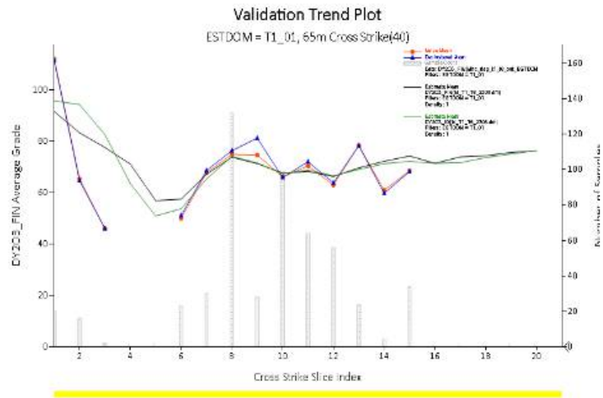


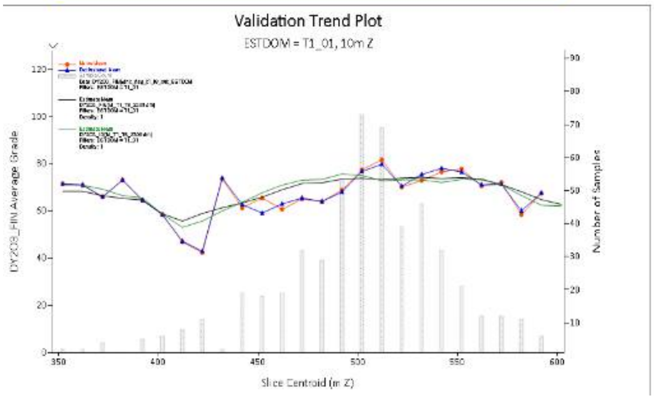


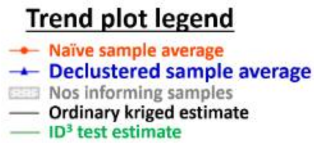


Dy_2O_3

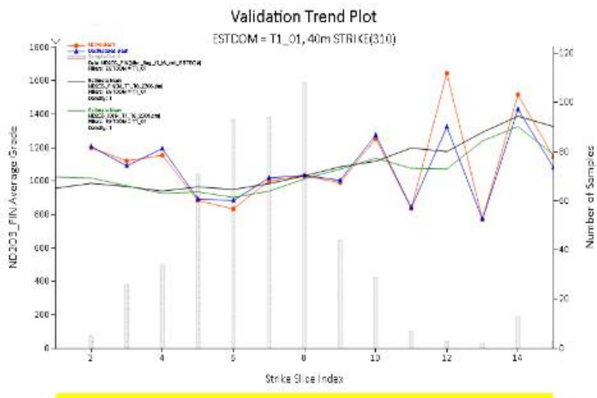

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
DY2O3_FIN Average Grade
Number of Samples
Strike Slice Index

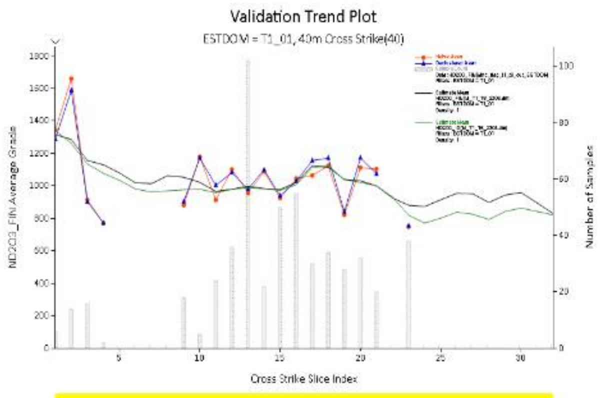

Validation Trend Plot
ESTDOM = T1_01, 65m Cross Strike(40)
DY2O3_FIN Average Grade
Cross Strike Slice Index

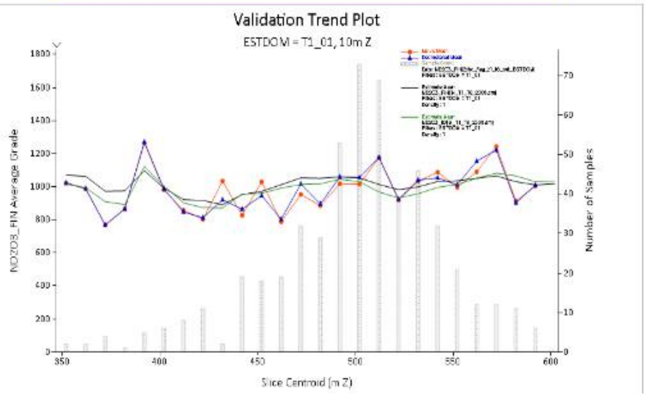

Validation Trend Plot
ESTDOM = T1_01, 10m Z
DY2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

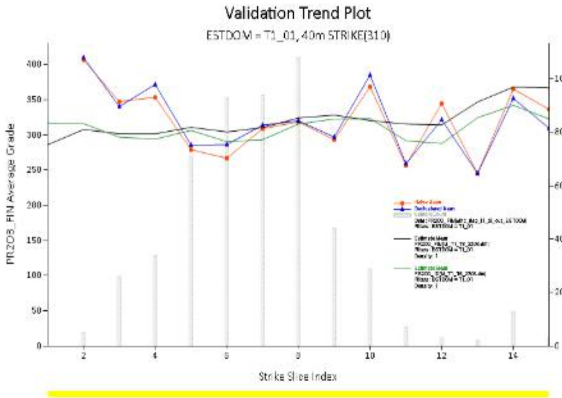

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
ND2O3_FIN Average Grade
Number of Samples
Strike Slice Index

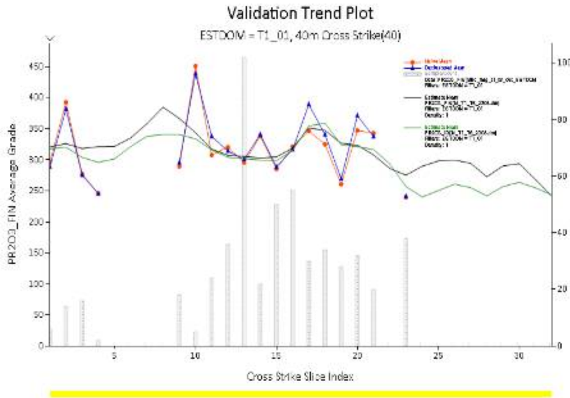

Validation Trend Plot
ESTDOM = T1_01, 40m Cross Strike(40)
ND2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

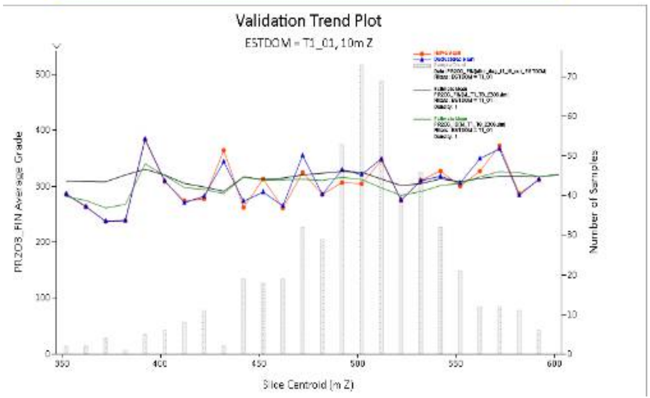

Validation Trend Plot
ESTDOM = T1_01, 10m Z
ND2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3

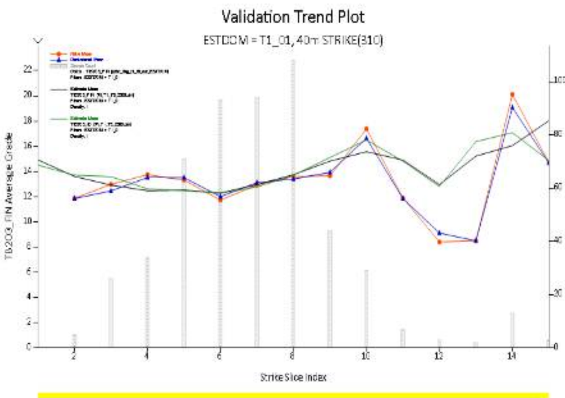

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
PR2O3_FIN Average Grade
Number of Samples
Strike Slice Index

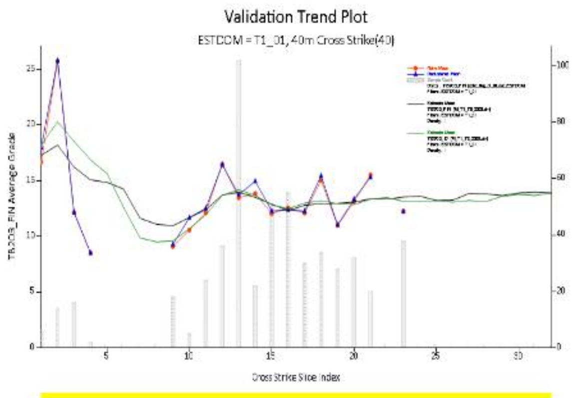

Validation Trend Plot
ESTDOM = T1_01, 40m Cross Strike(40)
PR2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

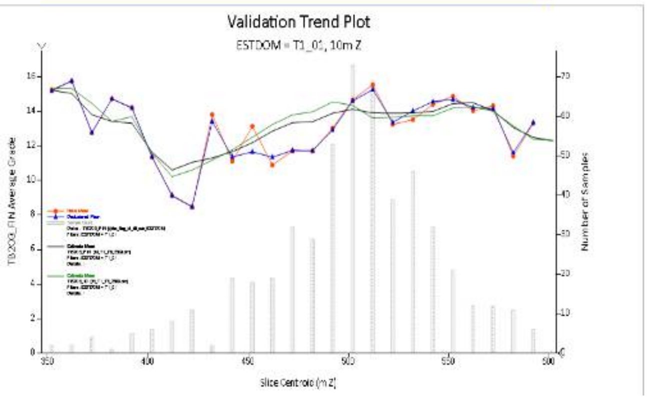

Validation Trend Plot
ESTDOM = T1_01, 10m Z
PR2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

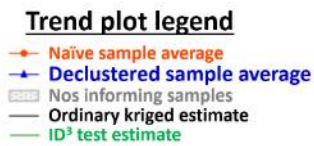

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Tb_2O_3

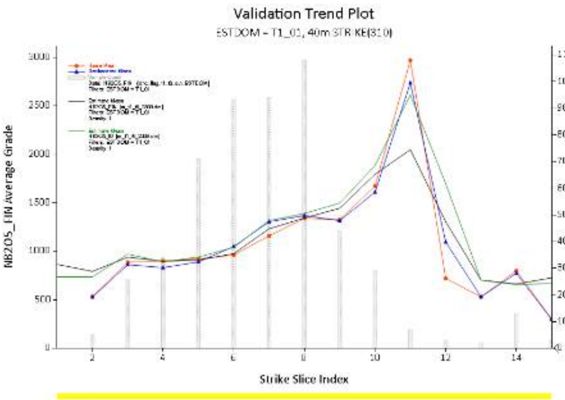

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
TB2O3_FIN Average Grade
Number of Samples
Strike Slice Index

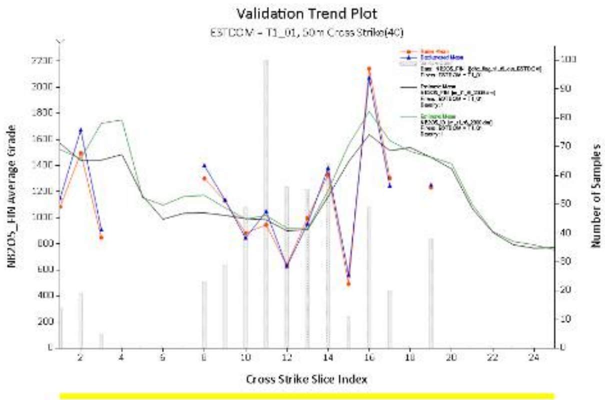

Validation Trend Plot
ESTDOM = T1_01, 40m Cross Strike(40)
TB2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

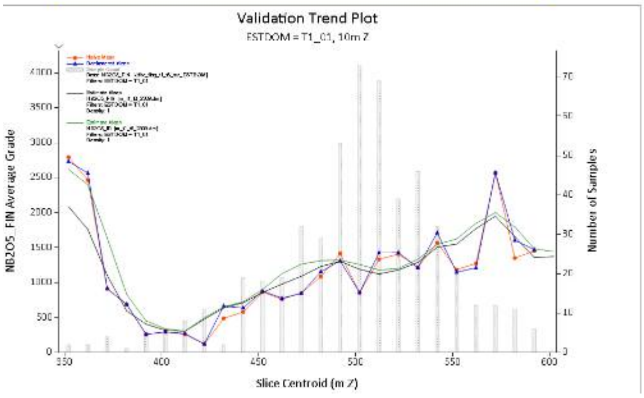

Validation Trend Plot
ESTDOM = T1_01, 10m Z
TB2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

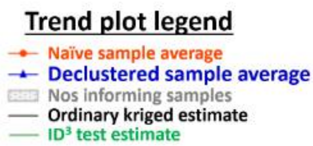

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nb_2O_5

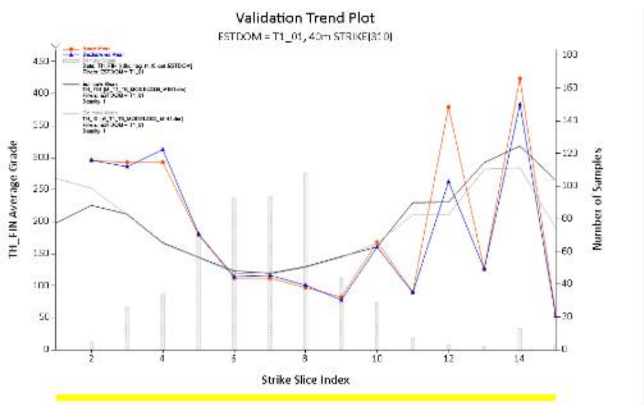

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
NB2O5_FIN Average Grade
Number of Samples
Strike Slice Index

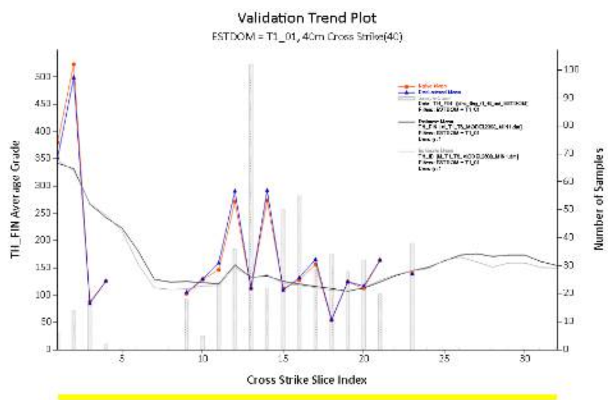

Validation Trend Plot
ESTDOM = T1_01, 50m Cross Strike(40)
NB2O5_FIN Average Grade
Number of Samples
Cross Strike Slice Index

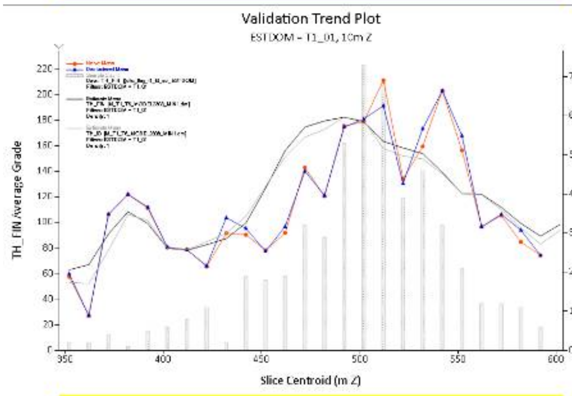

Validation Trend Plot
ESTDOM = T1_01, 10m Z
NB2O5_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

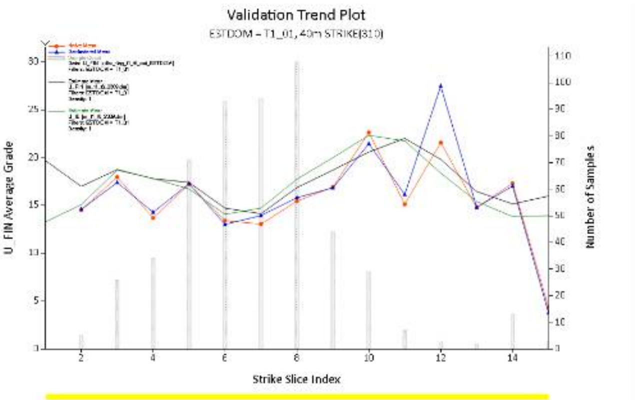

Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
TH_FIN Average Grade
Number of Samples
Strike Slice Index

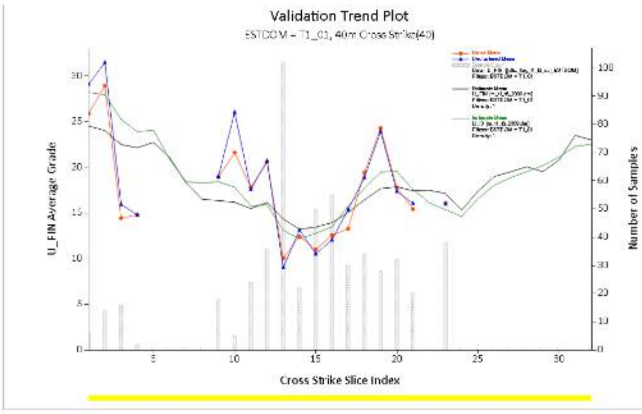

Validation Trend Plot
ESTDOM = T1_01, 40m Cross Strike(40)
TH_FIN Average Grade
Number of Samples
Cross Strike Slice Index

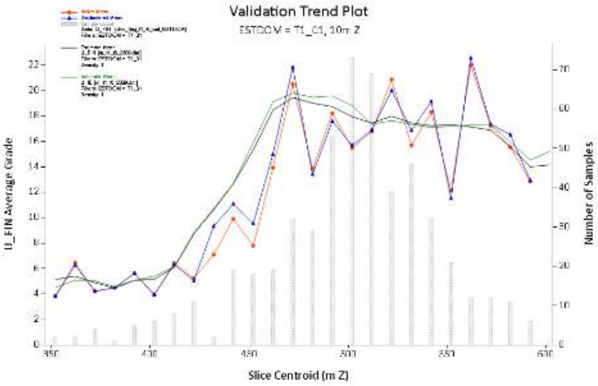

Validation Trend Plot
ESTDOM = T1_01, 10m Z
TH_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
ESTDOM = T1_01, 40m STRIKE(310)
U_FIN Average Grade
Number of Samples
Strike Slice Index

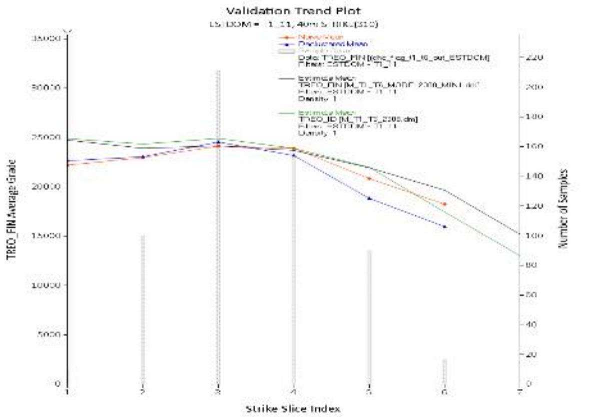

Validation Trend Plot
ESTDOM = T1_01, 40m Cross Strike(40)
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index

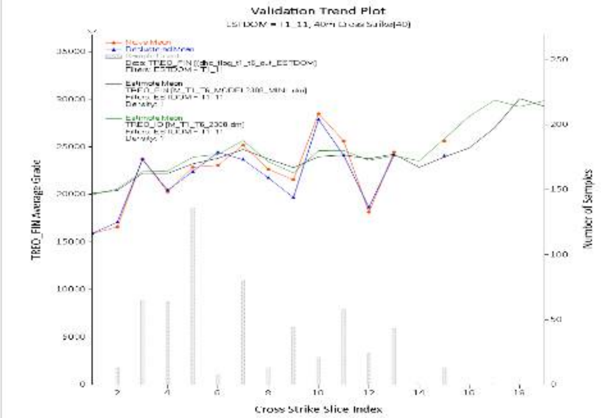

Validation Trend Plot
ESTDOM = T1_01, 10m Z
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

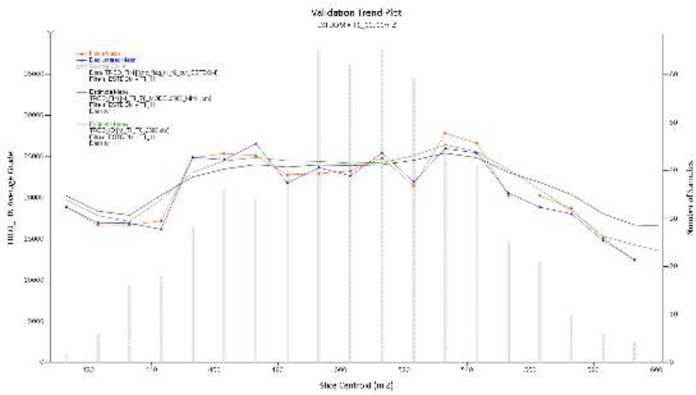

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

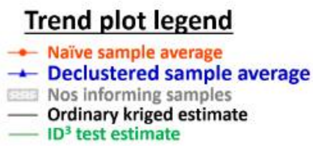
Model validation trend plots target 1 - T1_11

TREO

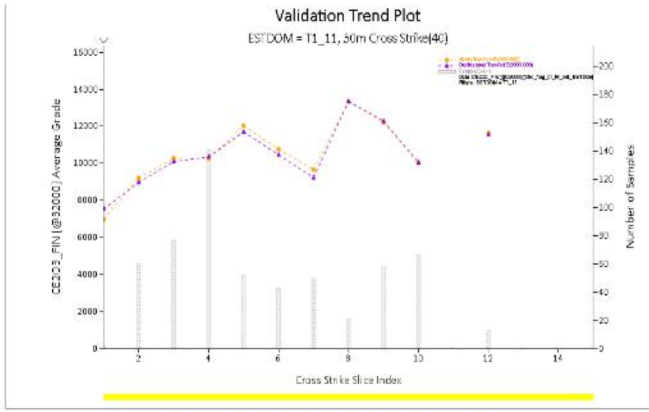


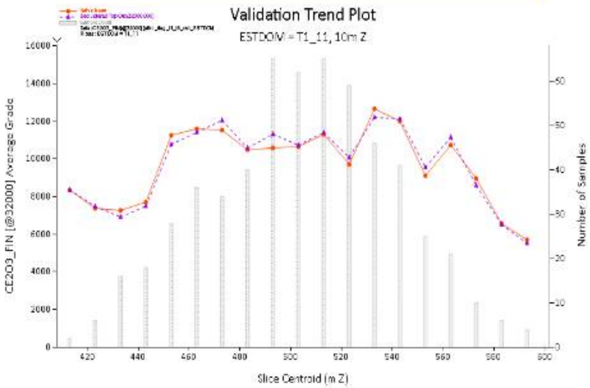


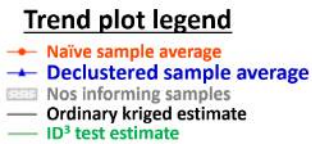




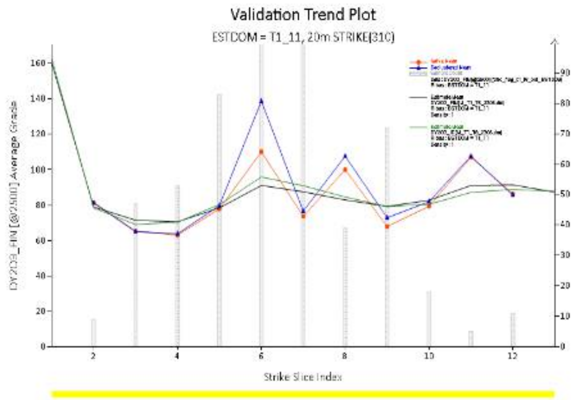
Ce_2O_3

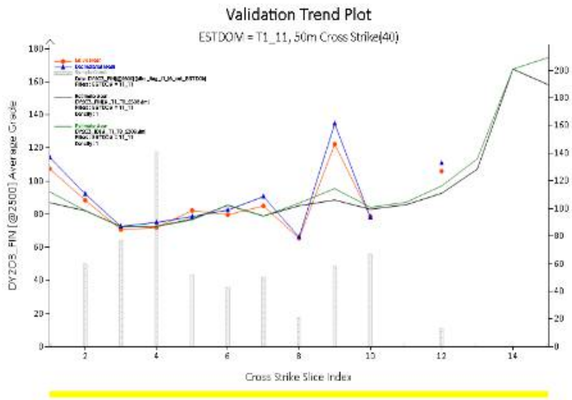


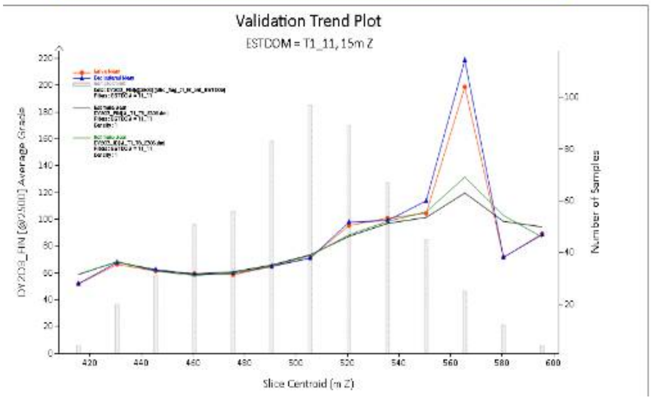


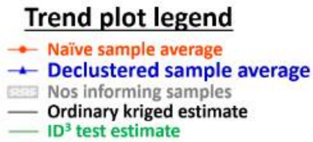


Dy_2O_3

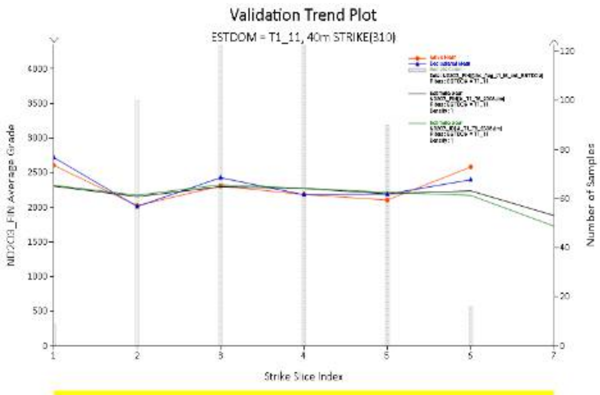

Validation Trend Plot
ESTDOM = T1_11, 20m STRIKE(310)
DY2O3_FIN [@2500] Average Grade
Number of Samples
Strike Slice Index

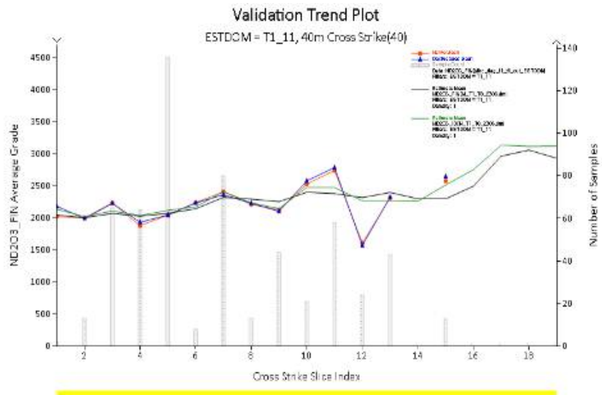

Validation Trend Plot
ESTDOM = T1_11, 50m Cross Strike(40)
DY2O3_FIN [@2500] Average Grade
Number of Samples
Cross Strike Slice Index

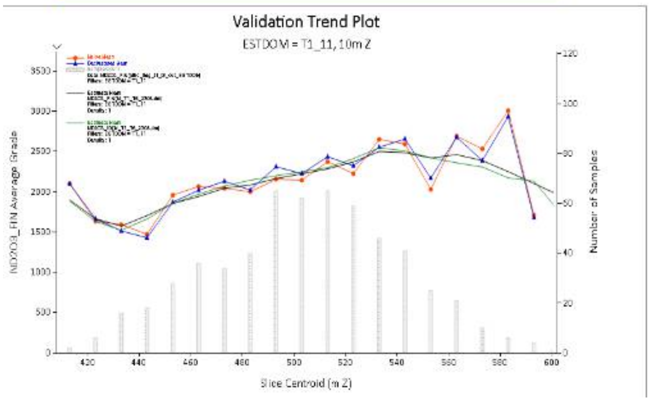

Validation Trend Plot
ESTDOM = T1_11, 15m Z
DY2O3_FIN [@2500] Average Grade
Number of Samples
Slice Centroid (m Z)

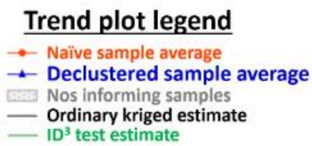

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID3 test estimate

Nd_2O_3

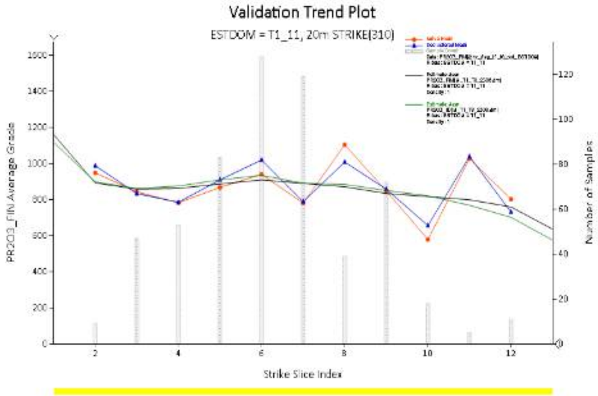

Validation Trend Plot
ESTDOM = T1_11, 40m STRIKE(310)
ND2O3_FIN Average Grade
Number of Samples
Strike Slice Index

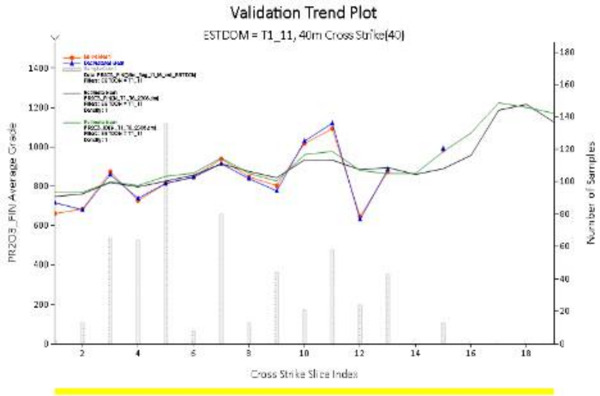

Validation Trend Plot
ESTDOM = T1_11, 40m Cross Strike(40)
ND2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

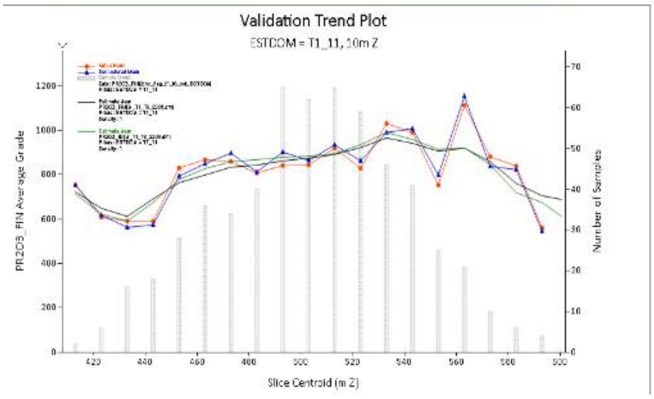

Validation Trend Plot
ESTDOM = T1_11, 10m Z
ND2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID3 test estimate

Pr_2O_3



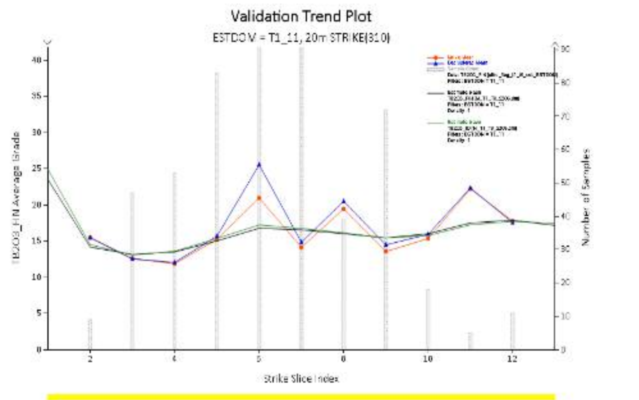


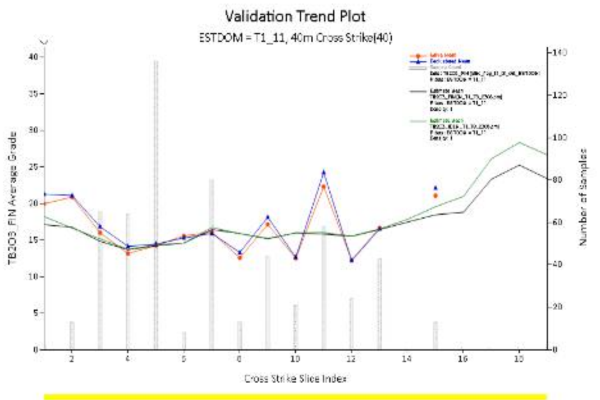


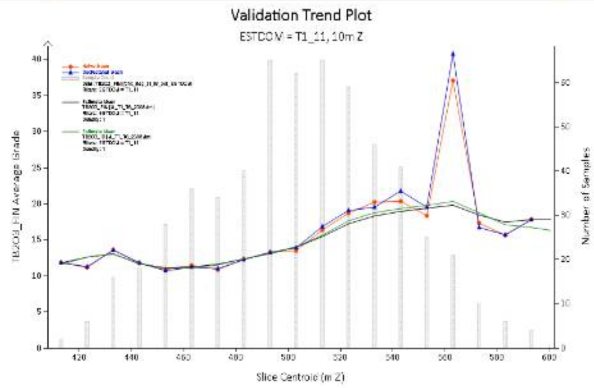
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Tb_2O_3



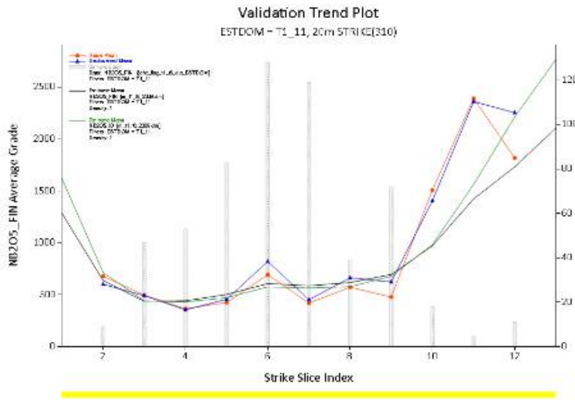


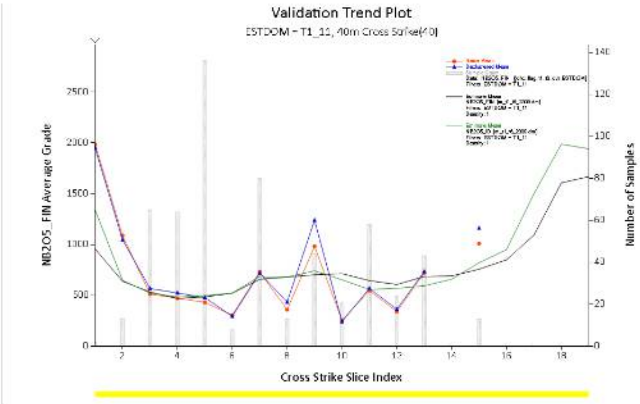


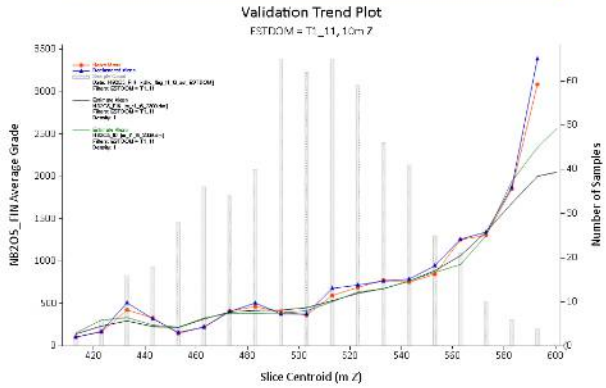
Trend plot legend

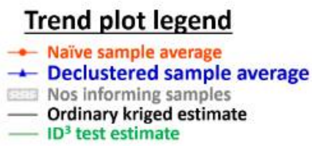
- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Nb_2O_5

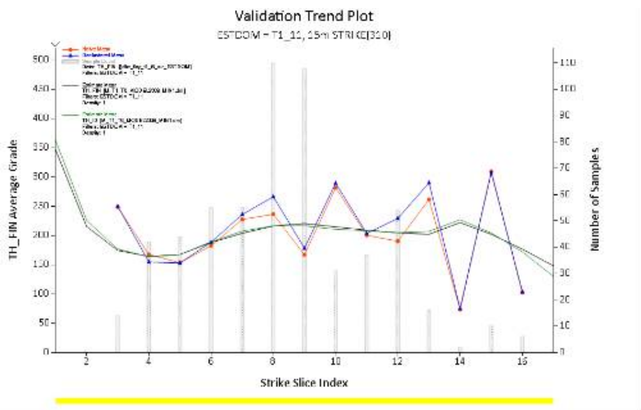

Validation Trend Plot
ESTDOM – T1_11, 20m STRIKE(310)
NB2O5_FIN Average Grade
Number of Samples
Strike Slice Index

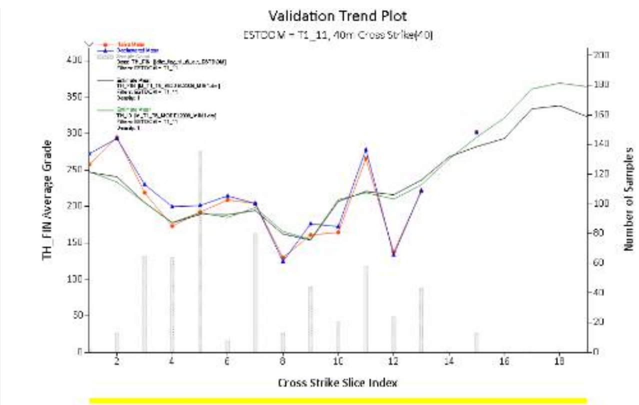

Validation Trend Plot
NB2O5_FIN Average Grade
Number of Samples
Cross Strike Slice Index

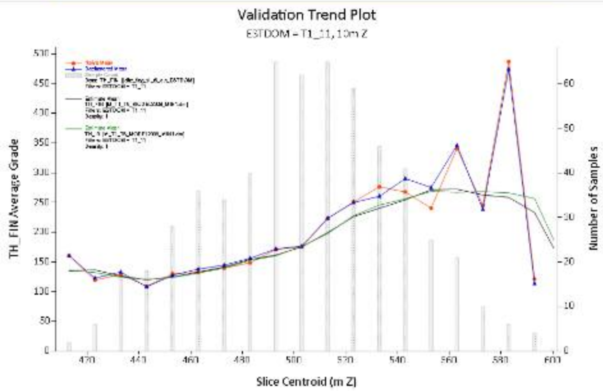

Validation Trend Plot
ESTDOM = T1_11, 10m Z
NB2O5_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

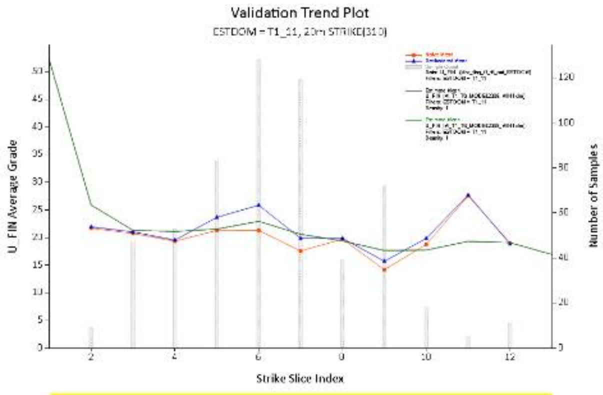

Validation Trend Plot
TH_FIN Average Grade
Number of Samples
Strike Slice Index

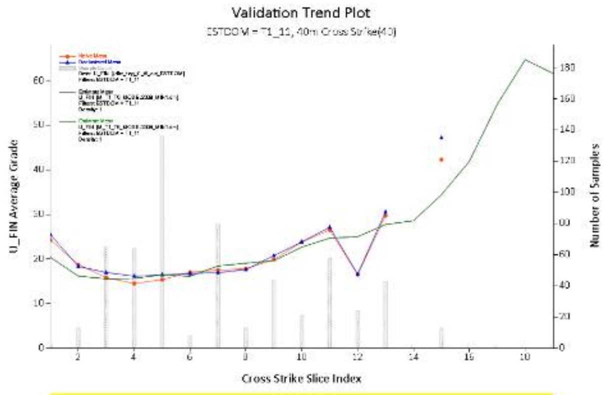

Validation Trend Plot
TH_FIN Average Grade
Number of Samples
Cross Strike Slice Index

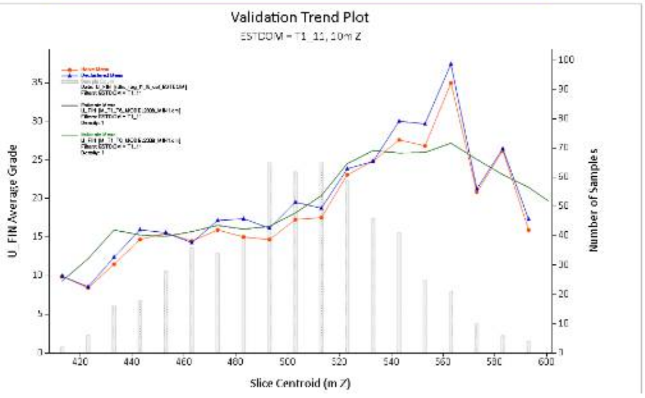

Validation Trend Plot
ESTDOM – T1_11, 10m Z
TH_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U





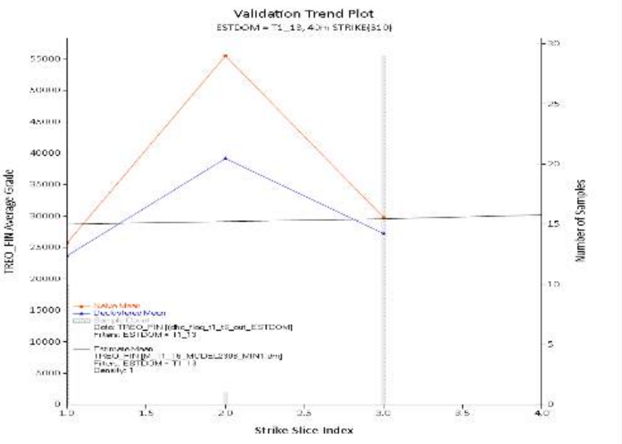


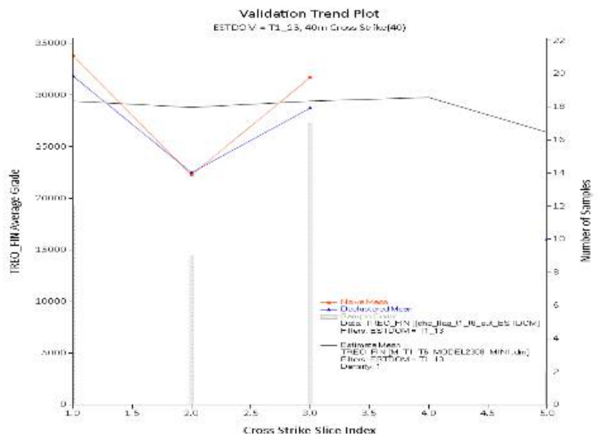
Trend plot legend

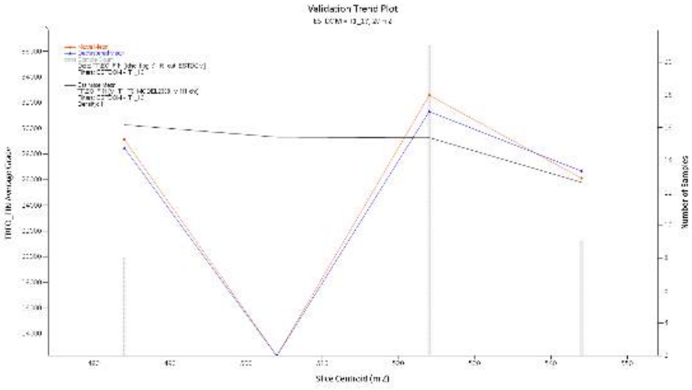
- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

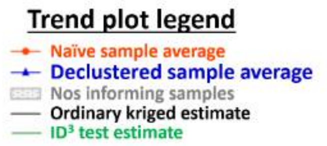
Model validation trend plots target 1 - T1_13

TREO

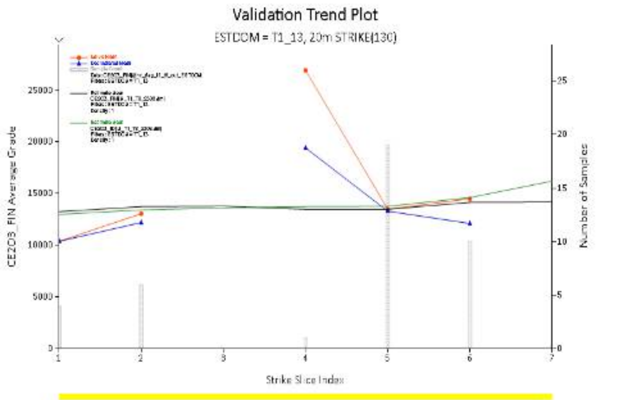


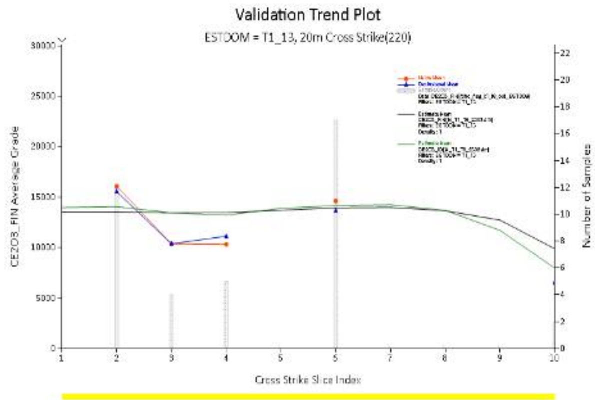


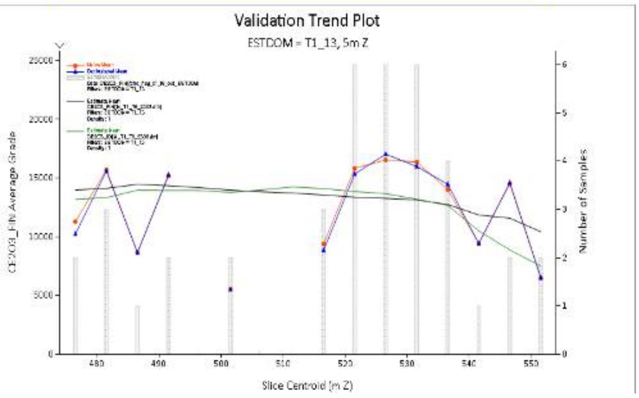




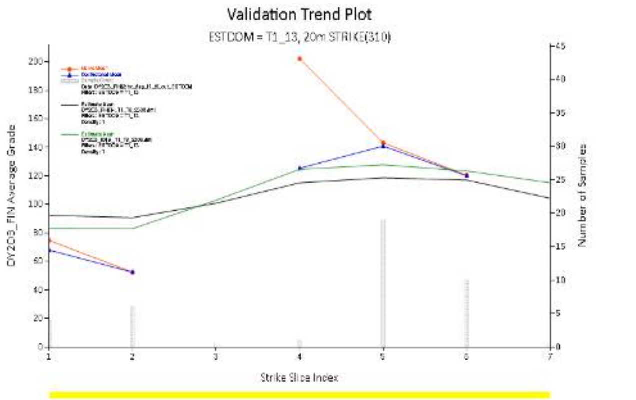
Ce_2O_3

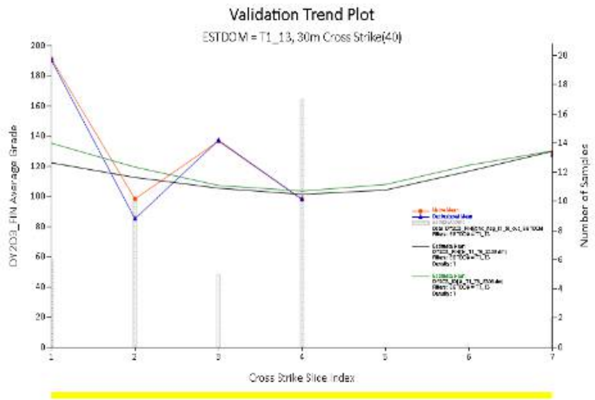


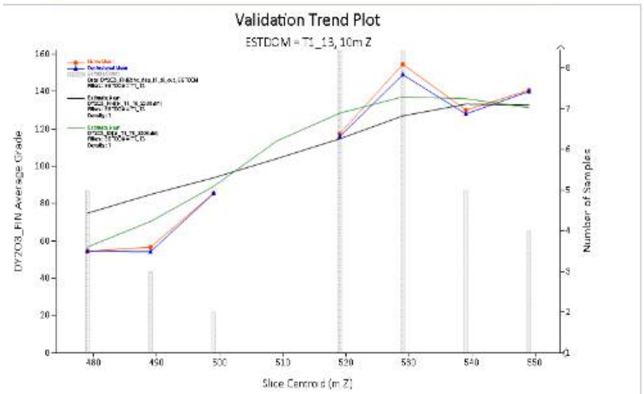


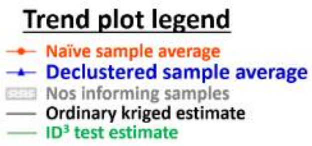


Dy_2O_3

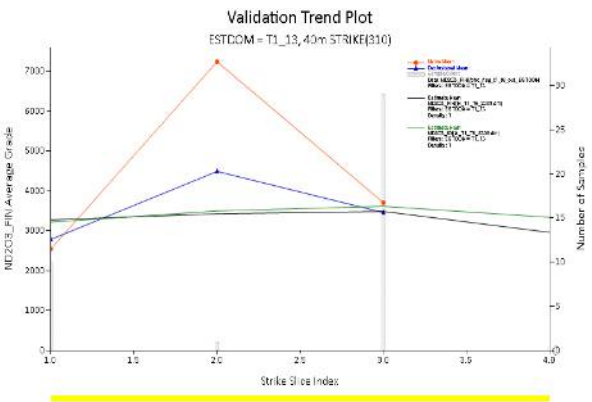

Validation Trend Plot
ESTDOM = T1_13, 20m STRIKE(310)
DY2O3_FIN Average Grade
Number of Samples
Strike Slice Index

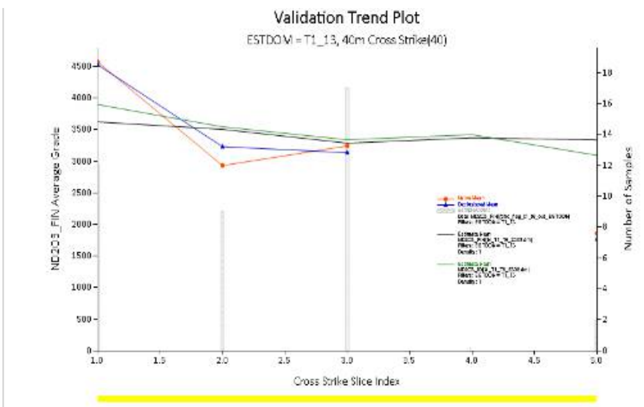

Validation Trend Plot
ESTDOM = T1_13, 30m Cross Strike(40)
DY2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

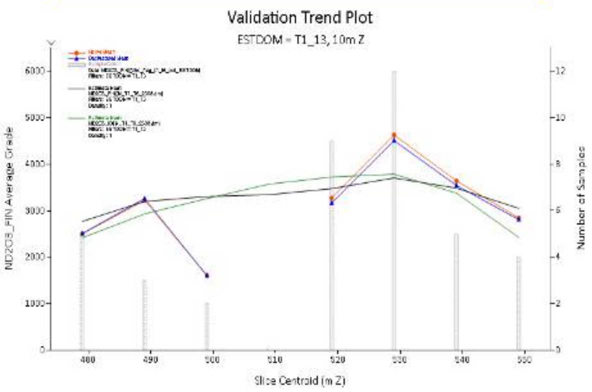

Validation Trend Plot
ESTDOM = T1_13, 10m Z
DY2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

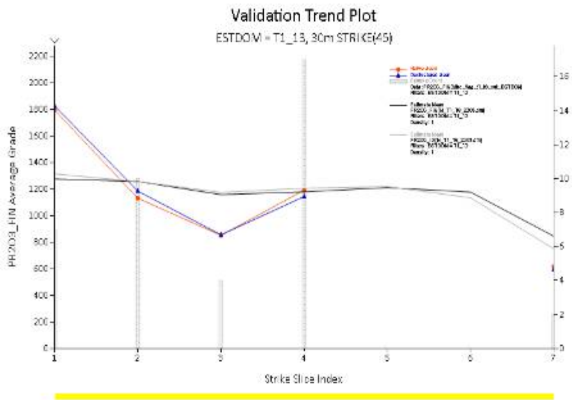

Validation Trend Plot
ESTDOM = T1_13, 40m STRIKE(310)
ND2O3_FIN Average Grade
Number of Samples
Strike Slice Index

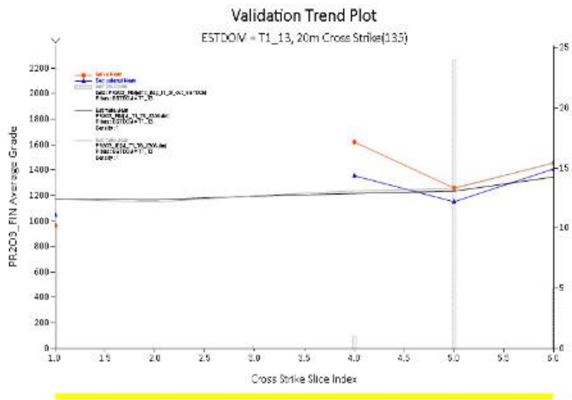

Validation Trend Plot
ESTDOM = T1_13, 40m Cross Strike(40)
ND2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

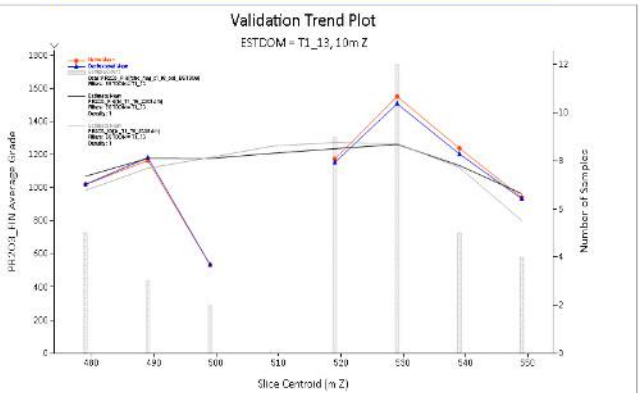

Validation Trend Plot
ESTDOM = T1_13, 10m Z
ND2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3



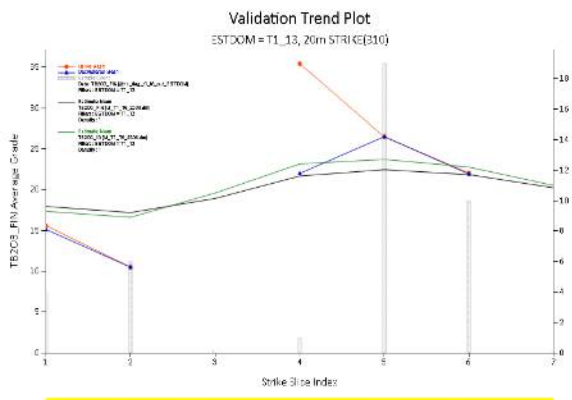


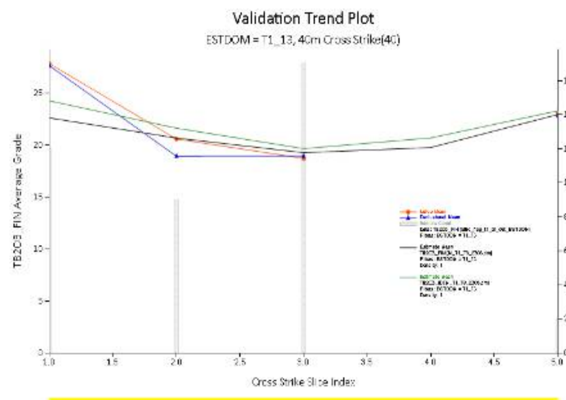


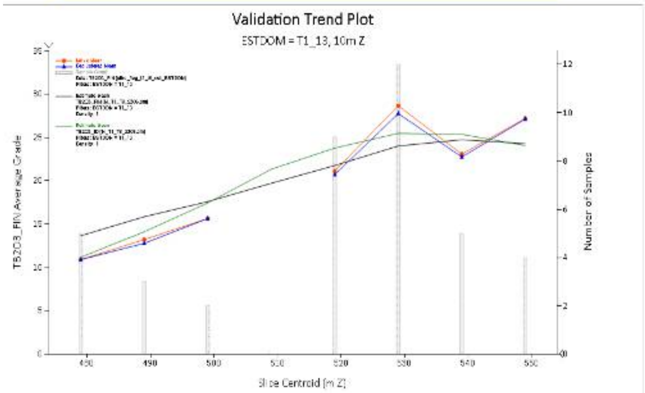
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Tb_2O_3



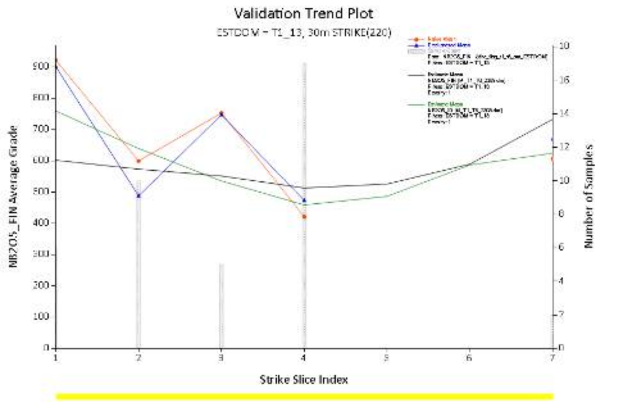


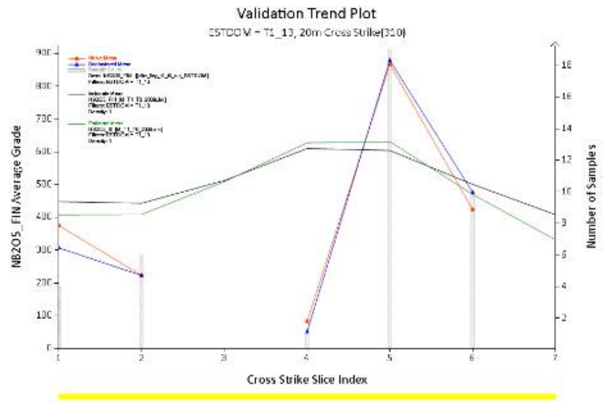


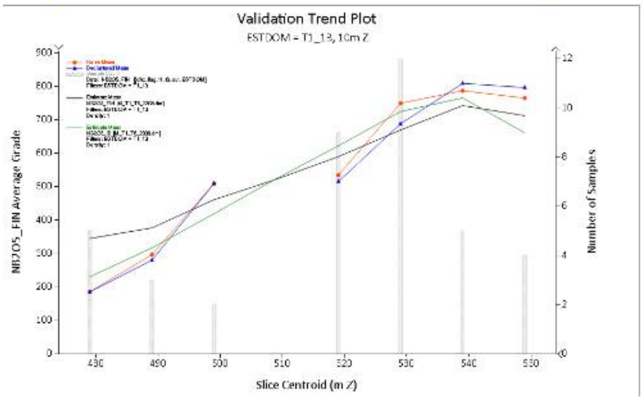
Trend plot legend

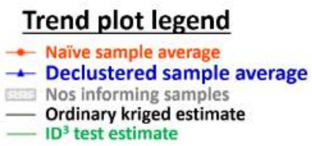
- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Nb_2O_5

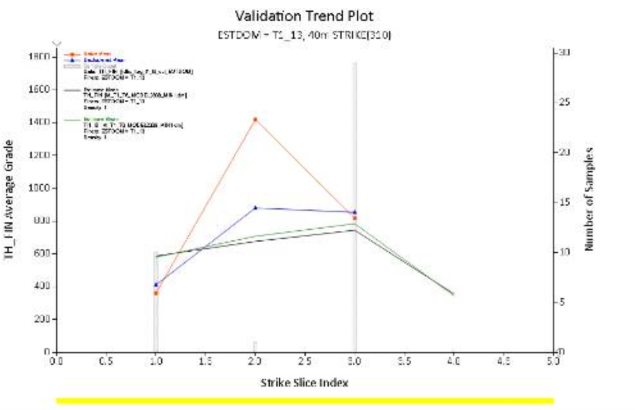

Validation Trend Plot
ESTDOM = T1_13, 30m STRIKE(220)
NB2O5_FIN Average Grade
Number of Samples
Strike Slice Index

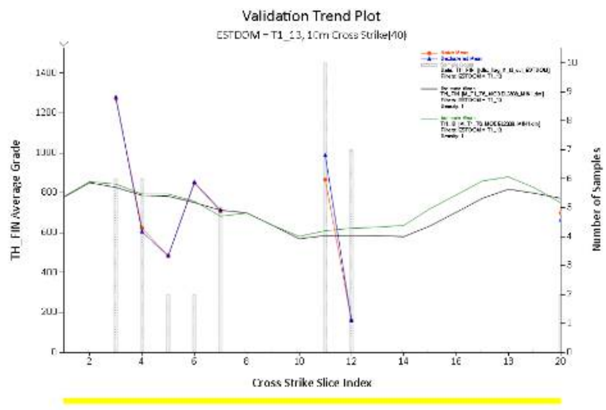

Validation Trend Plot
ESTDOM = T1_13, 20m Cross Strike(310)
NB2O5_FIN Average Grade
Number of Samples
Cross Strike Slice Index

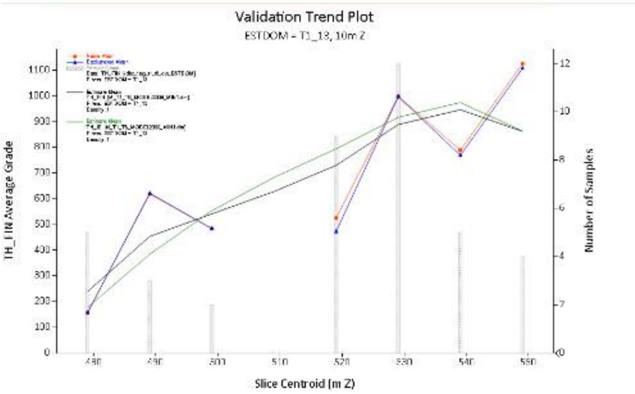

Validation Trend Plot
ESTDOM = T1_13, 10m Z
NB2O5_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

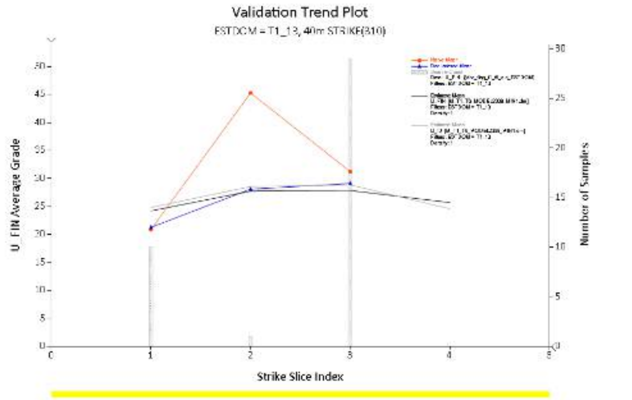

Validation Trend Plot
ESTDOM = T1_13, 40m STRIKE(310)
TH_FIN Average Grade
Number of Samples
Strike Slice Index

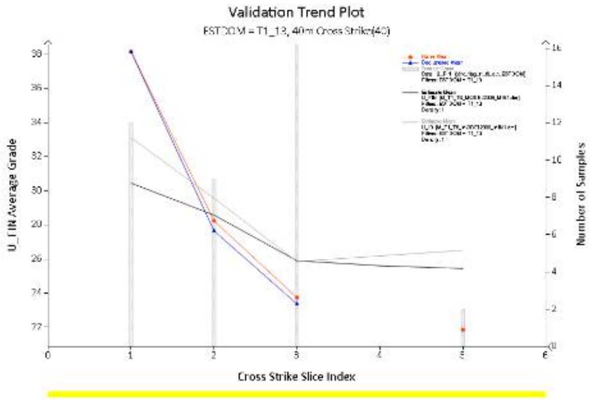

Validation Trend Plot
ESTDOM = T1_13, 10m Cross Strike(40)
TH_FIN Average Grade
Number of Samples
Cross Strike Slice Index

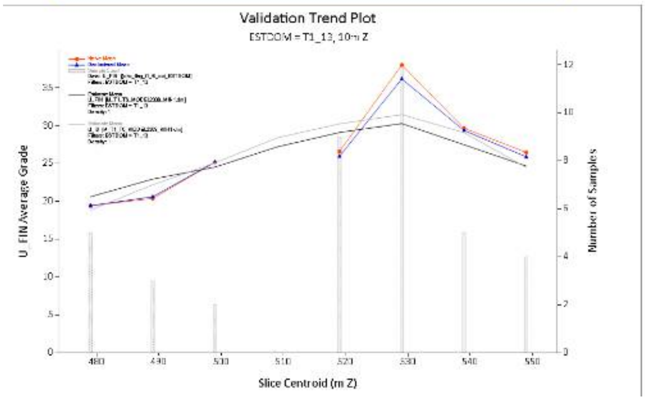

Validation Trend Plot
ESTDOM = T1_13, 10m Z
TH_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
ESTDOM = T1_13, 40m STRIKE(310)
U_FIN Average Grade
Number of Samples
Strike Slice Index

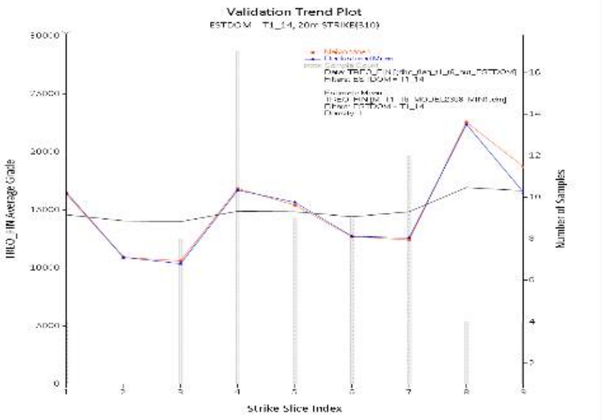

Validation Trend Plot
ESTDOM = T1_13, 40m Cross Strike(40)
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index

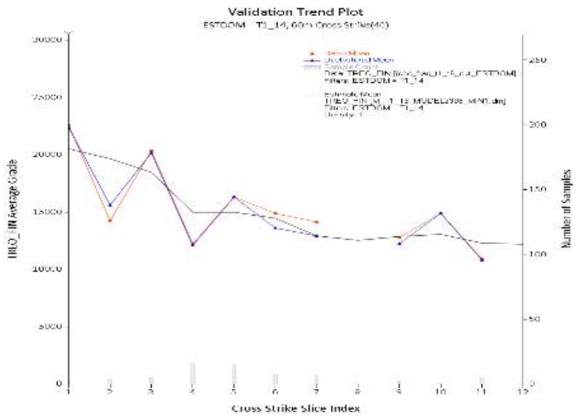

Validation Trend Plot
ESTDOM = T1_13, 10m Z
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

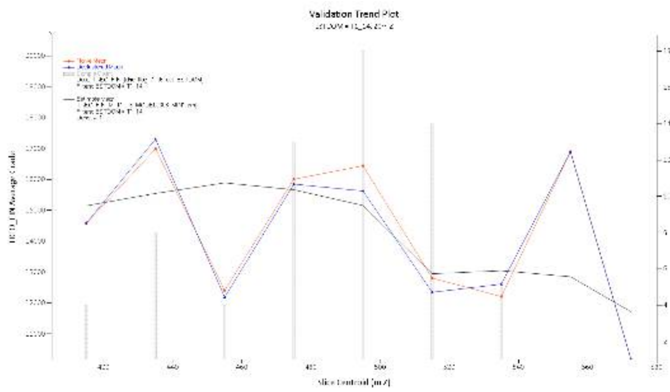

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Model validation trend plots target 1 - T1_14

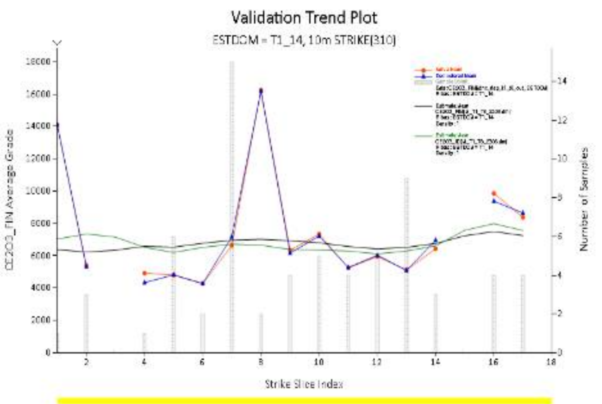
TREO

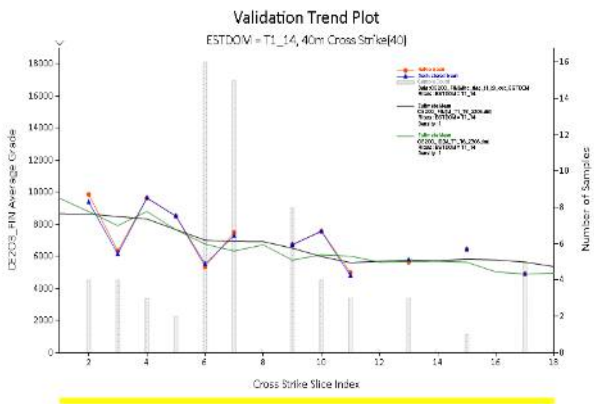


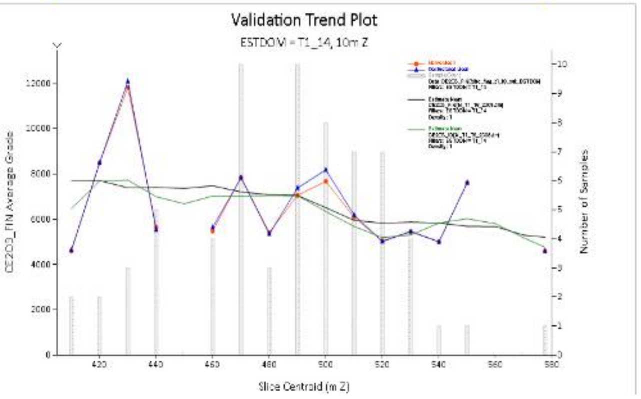




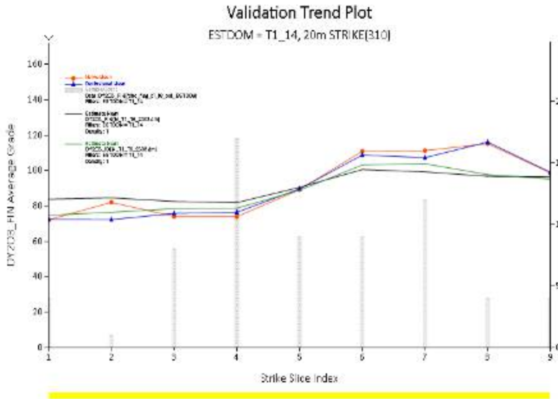
Ce_2O_3

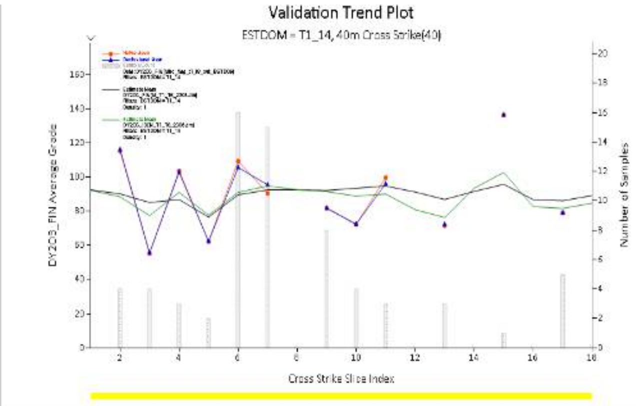


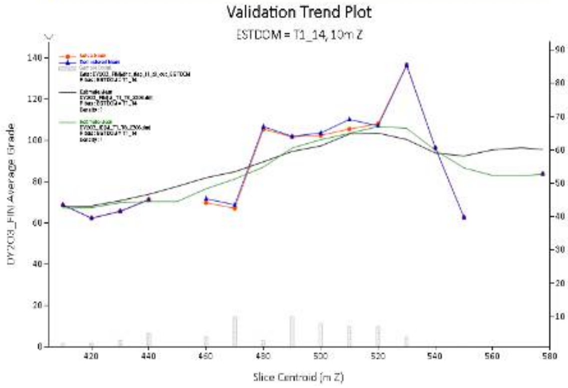




Dy_2O_3

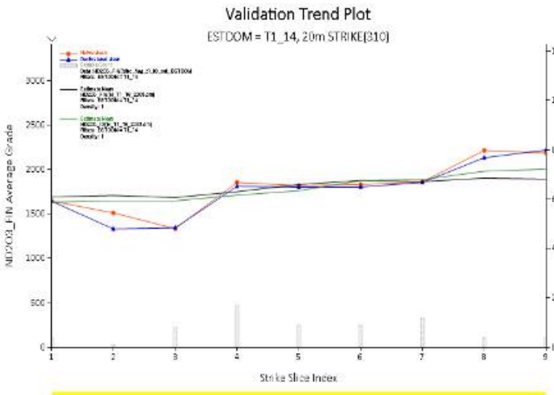

Validation Trend Plot
ESTDOM = T1_14, 20m STRIKE(310)
DY2O3_FIN Average Grade
Number of Samples
Strike Slice Index

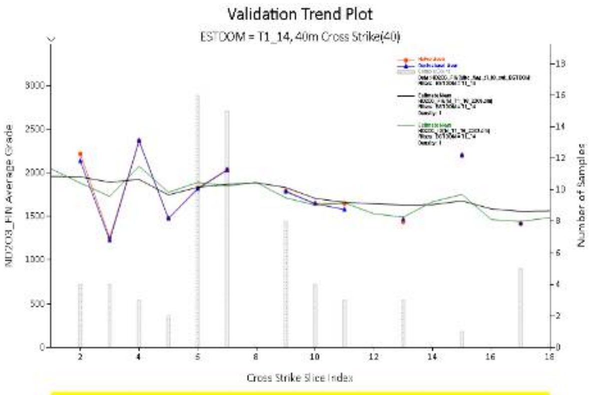

Validation Trend Plot
ESTDOM = T1_14, 40m Cross Strike(40)
DY2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

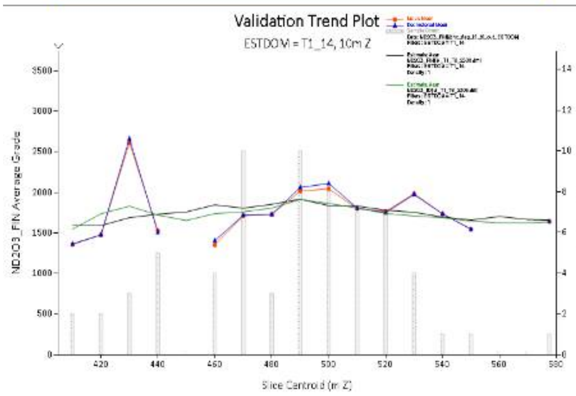

Validation Trend Plot
ESTDOM = T1_14, 10m Z
DY2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

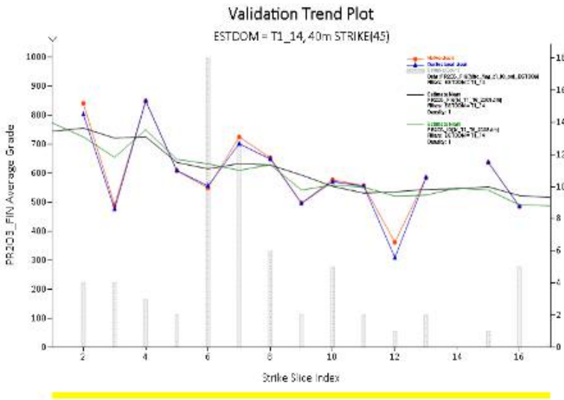

Validation Trend Plot
ESTDOM = T1_14, 20m STRIKE(310)
ND2O3_FIN Average Grade
Number of Samples
Strike Slice Index

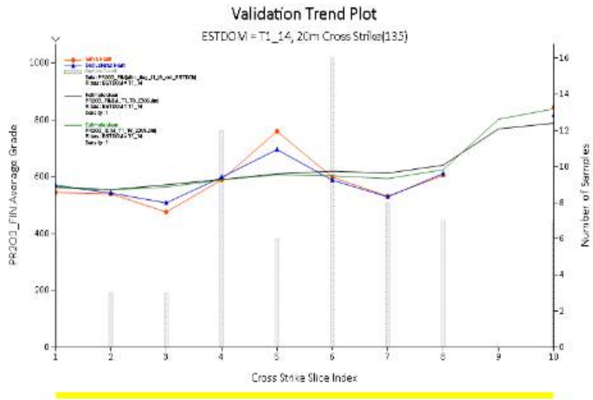

Validation Trend Plot
ESTDOM = T1_14, 40m Cross Strike(40)
ND2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

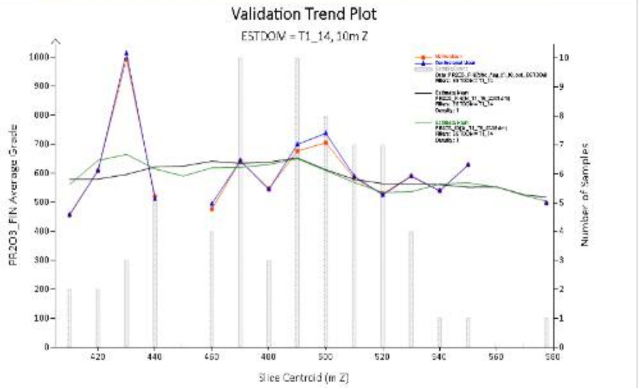

Validation Trend Plot
ESTDOM = T1_14, 10m Z
ND2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3



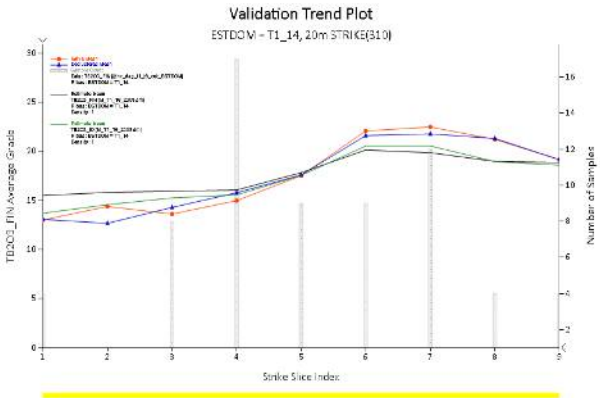


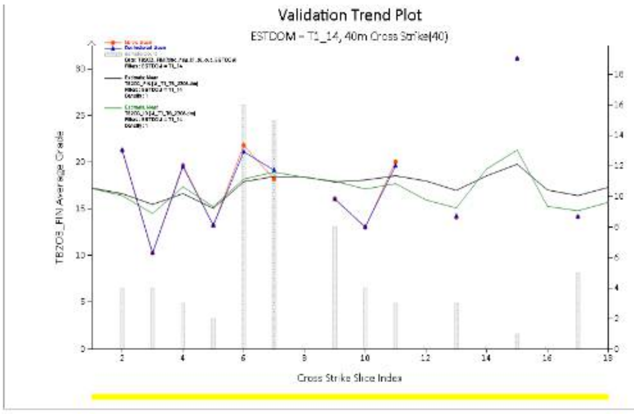


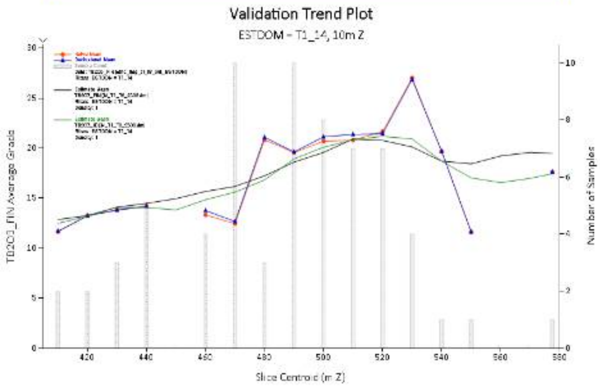
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Tb_2O_3



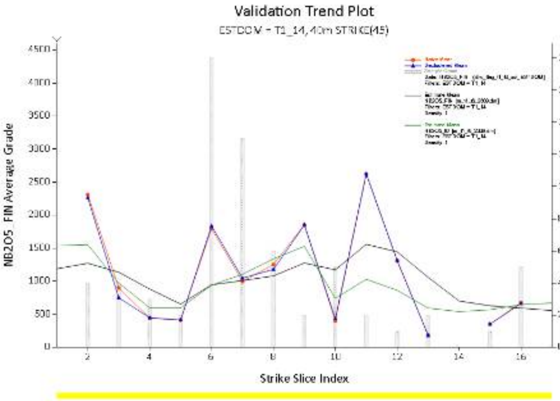


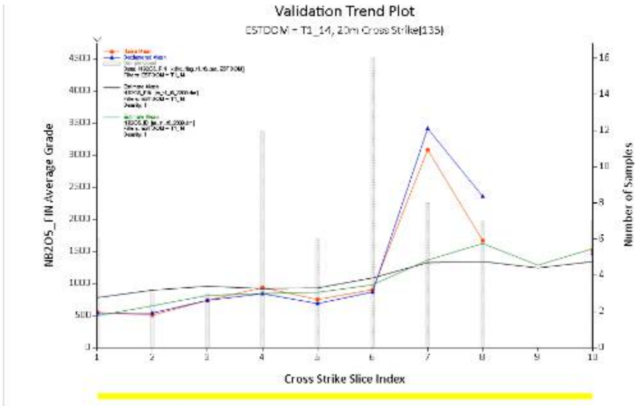


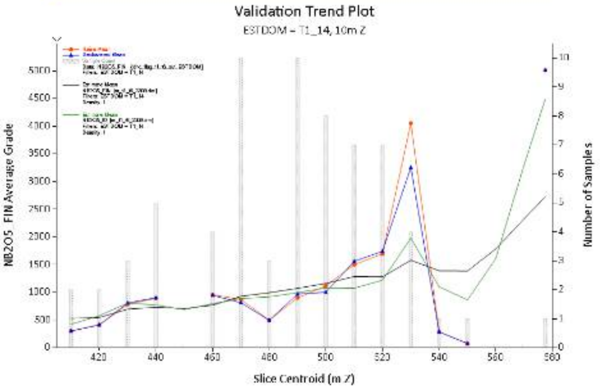
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Nb_2O_5



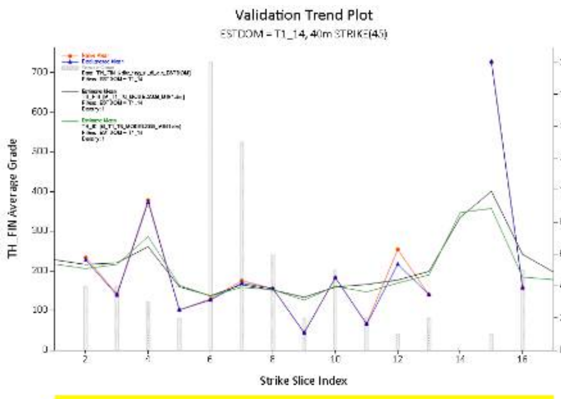


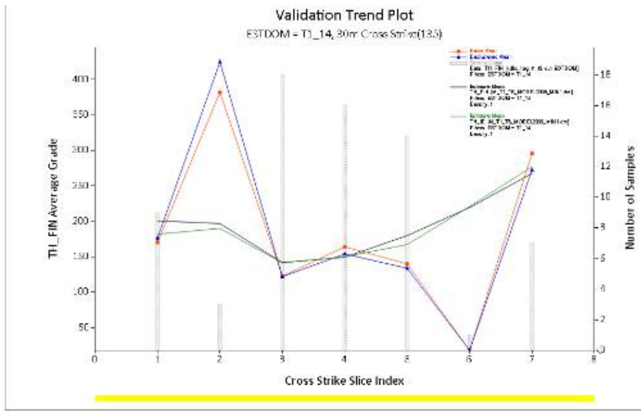


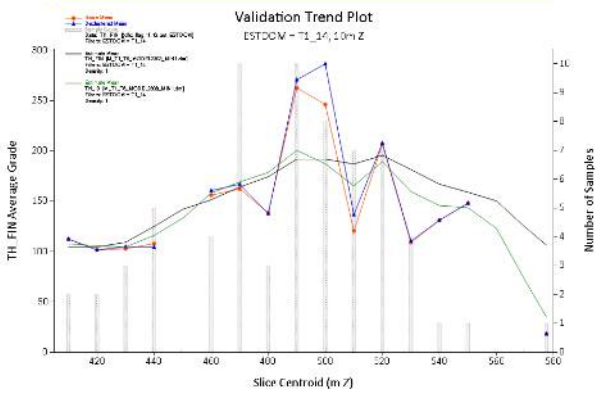
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Th



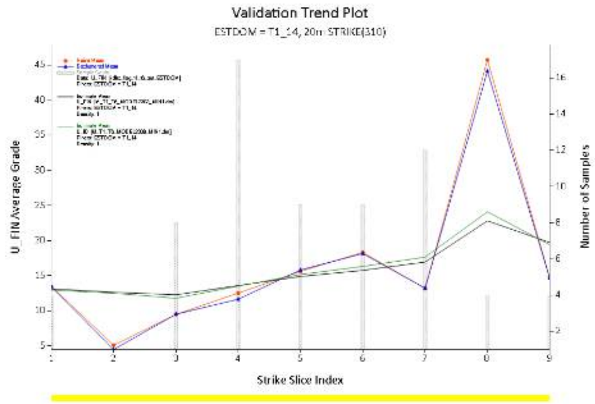


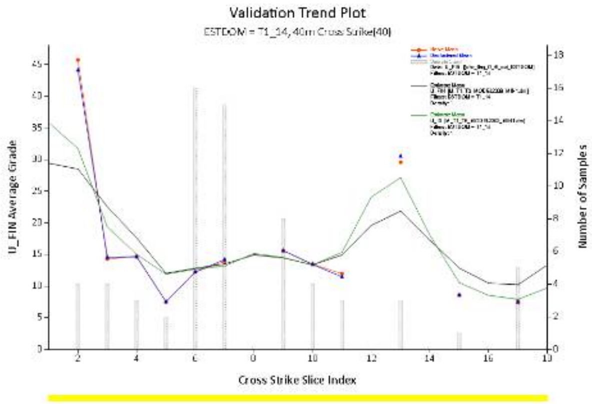


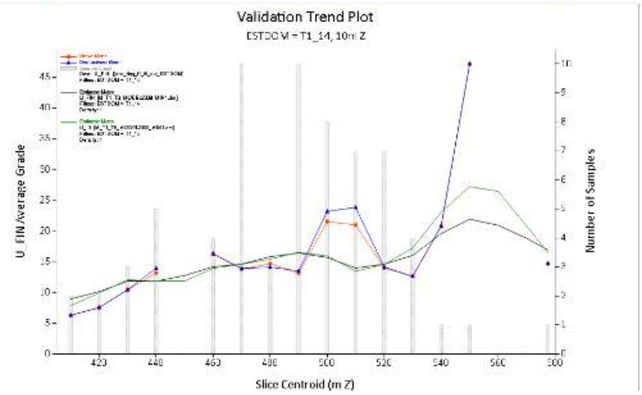
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

U


Validation Trend Plot
ESTDOM = T1_14, 20m STRIKE(310)
U_FIN Average Grade
Number of Samples
Strike Slice Index

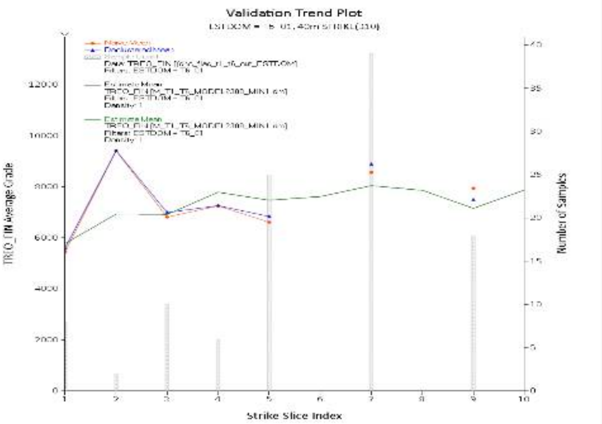

Validation Trend Plot
ESTDOM = T1_14, 40m Cross Strike(40)
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index

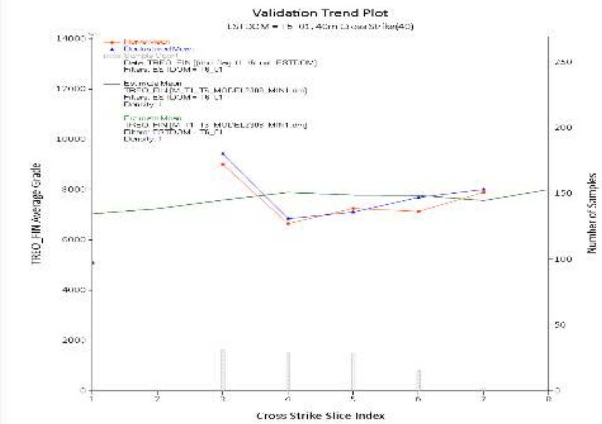

Validation Trend Plot
ESTDOM = T1_14, 10m Z
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

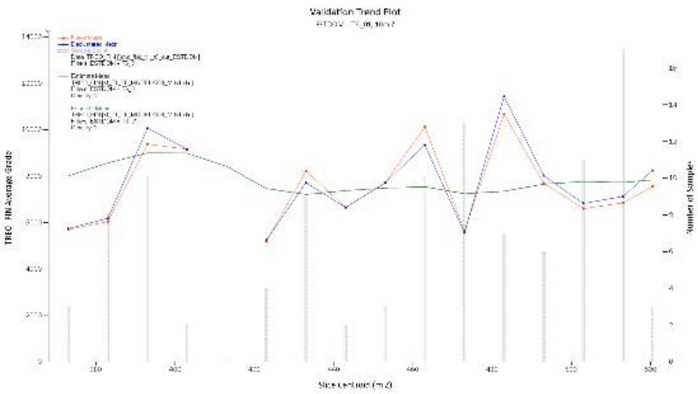

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

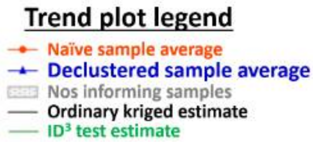
Model validation trend plots target 6 – T6_01

TREO

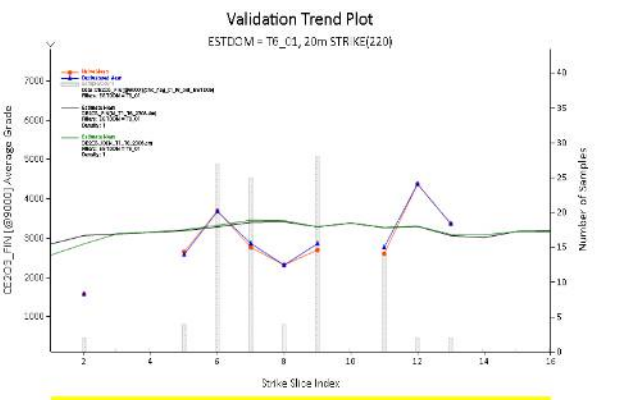


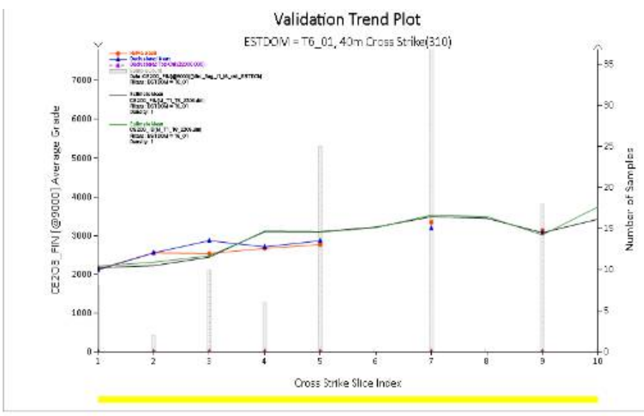


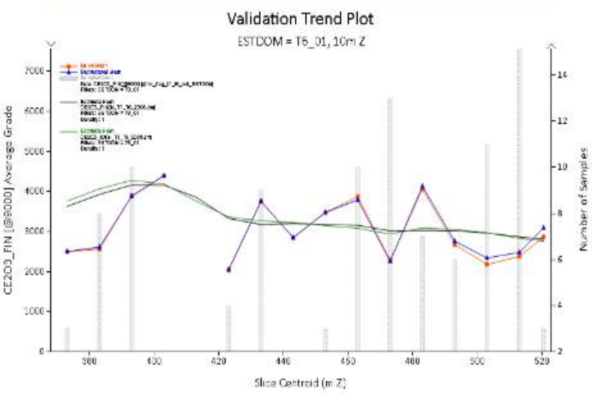




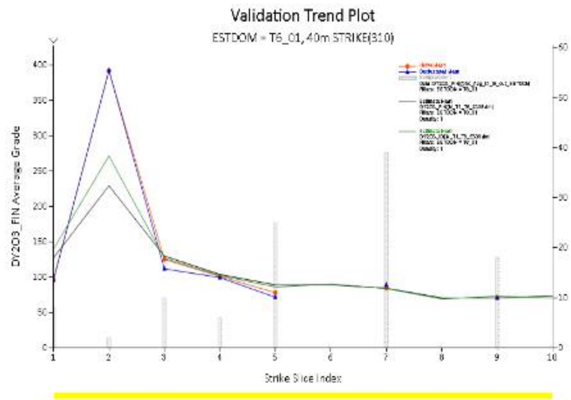
Ce_2O_3

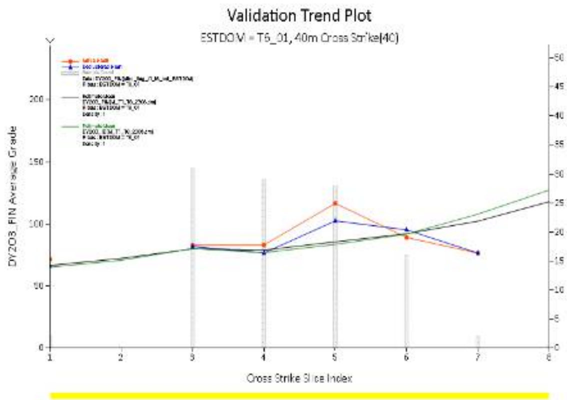


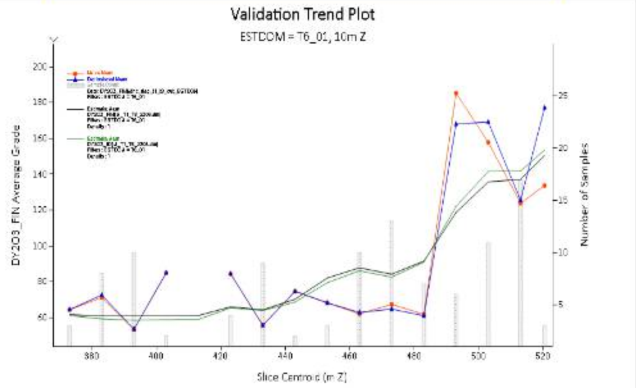




Dy_2O_3



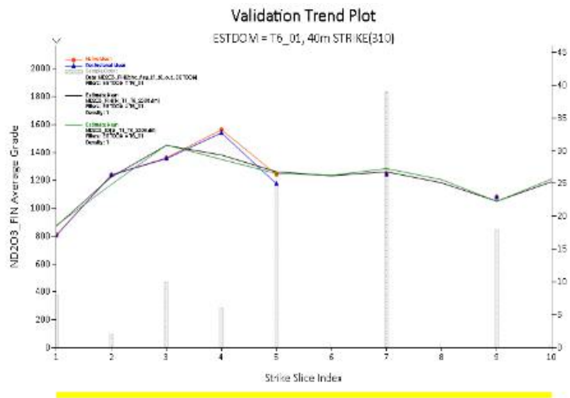


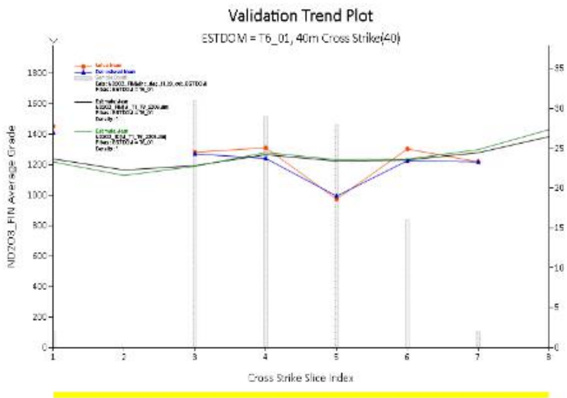


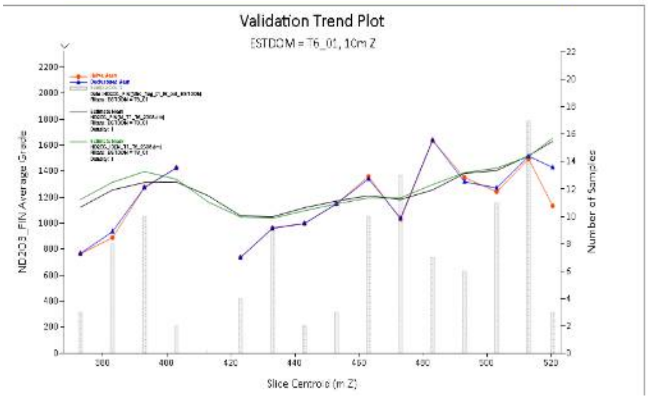
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Nd_2O_3



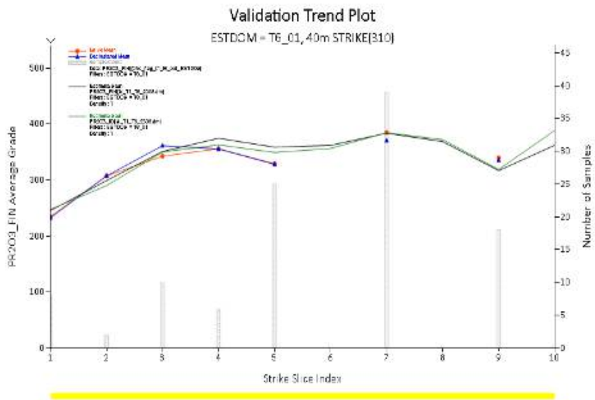


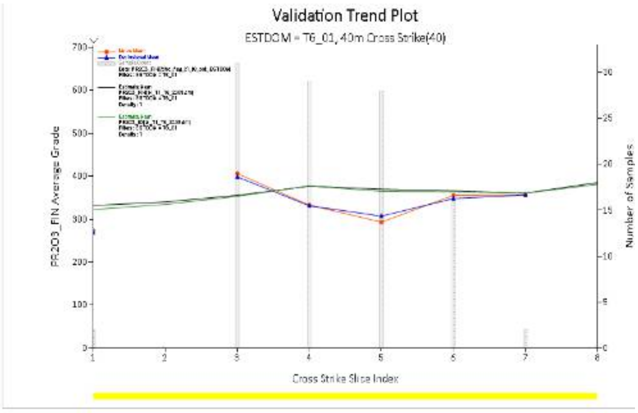


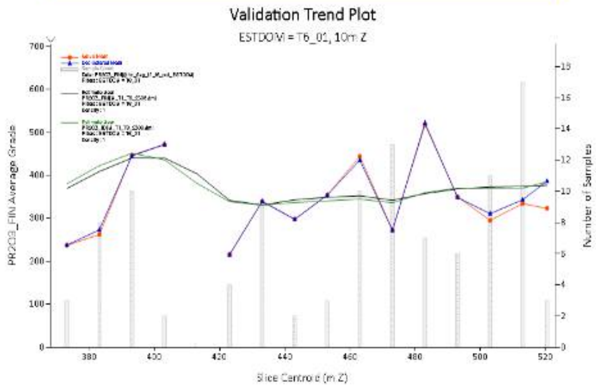
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Pr_2O_3



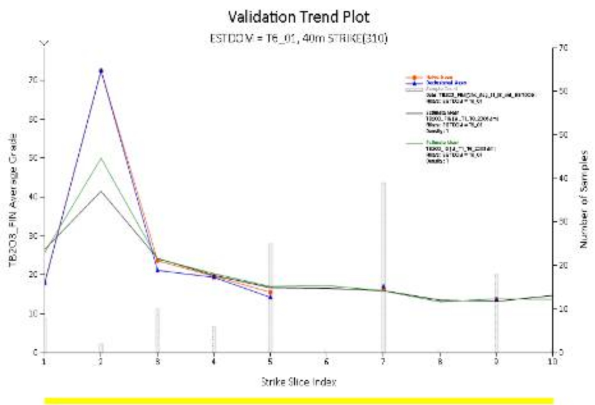


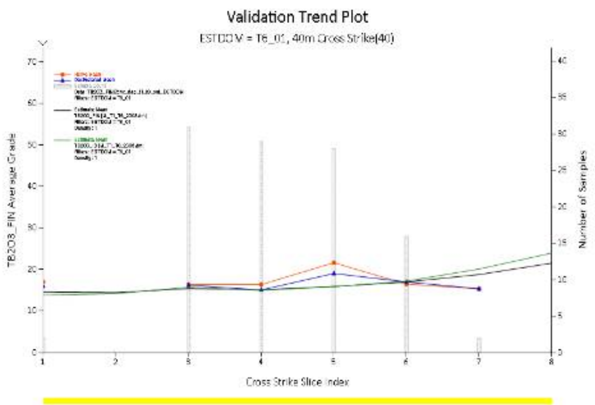


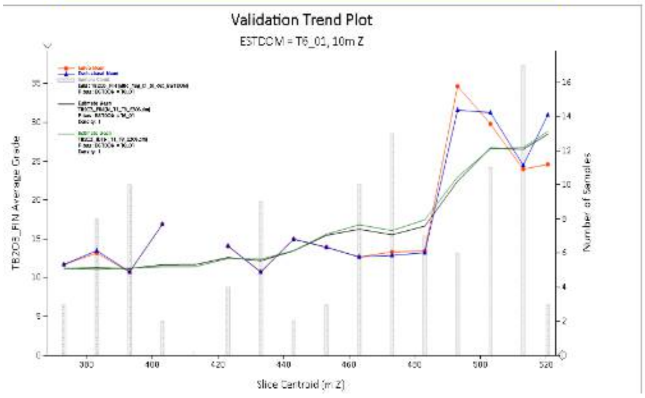
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Tb_2O_3



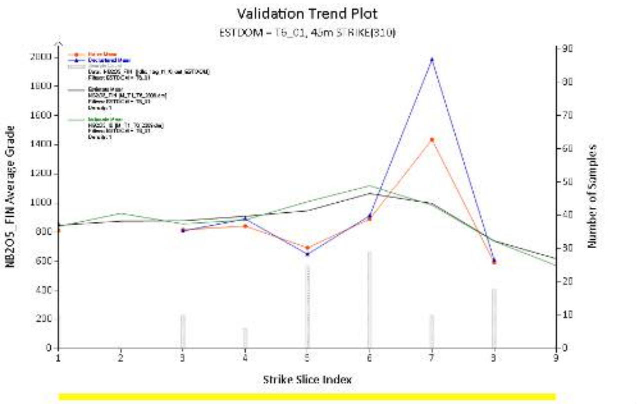


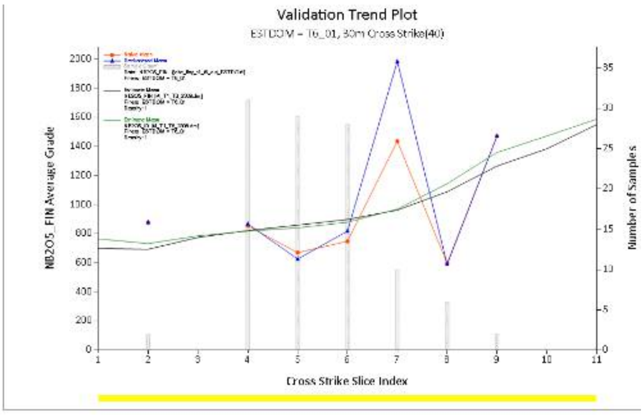


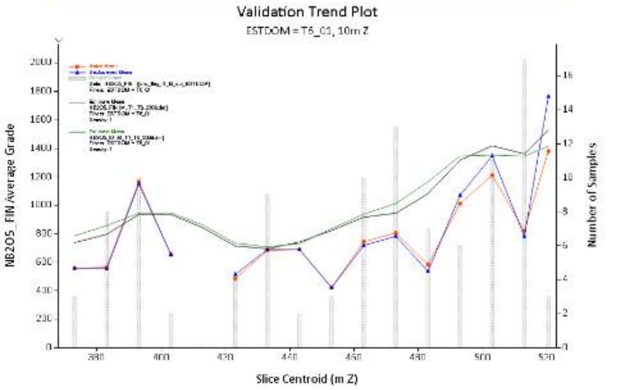
Trend plot legend

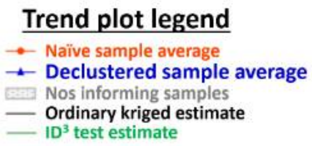
- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID^3 test estimate

Nb_2O_5

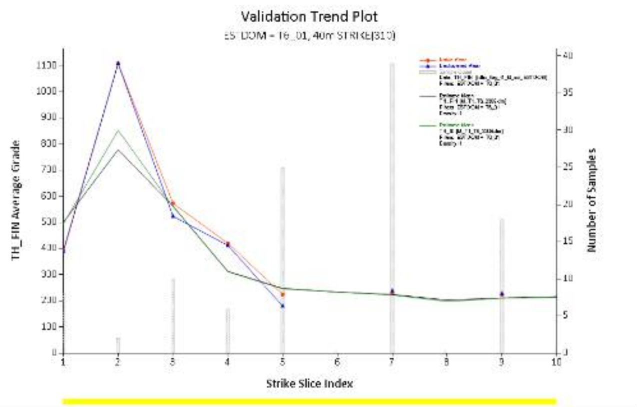

Validation Trend Plot
ESTDOM = T6_01, 45m STRIKE(310)
NB2O5_FIN Average Grade
Number of Samples
Strike Slice Index

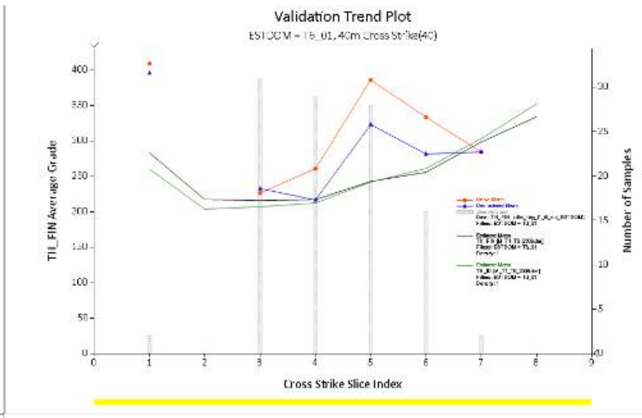

Validation Trend Plot
ESTDOM = T6_01, 30m Cross Strike(40)
NB2O5_FIN Average Grade
Number of Samples
Cross Strike Slice Index

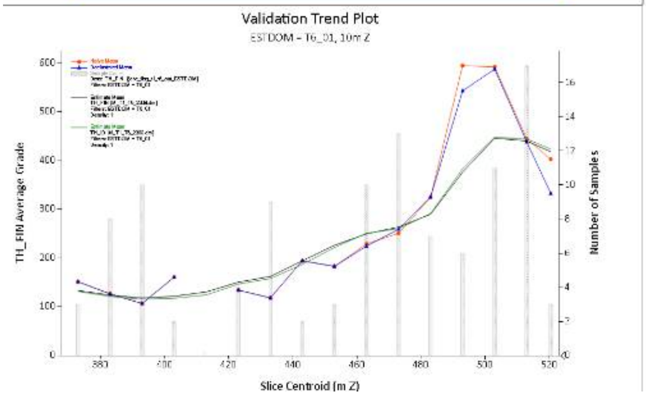

Validation Trend Plot
ESTDOM = T6_01, 10m Z
NB2O5_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

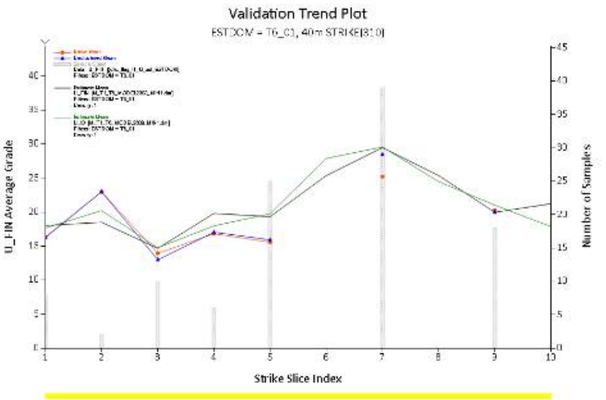

Validation Trend Plot
ESTDOM = T6_01, 40m STRIKE(310)
TH_FIN Average Grade
Number of Samples
Strike Slice Index

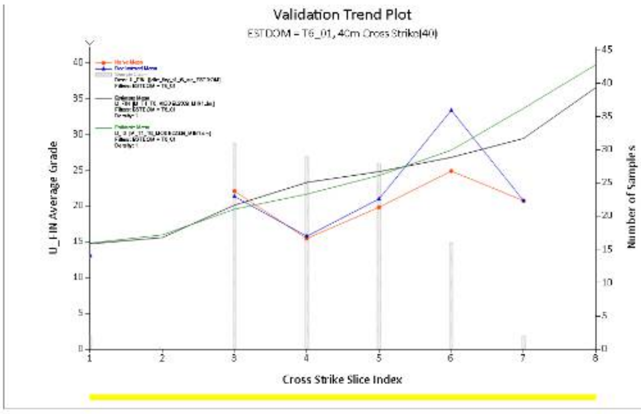

Validation Trend Plot
ESTDOM = T6_01, 40m Cross Strike(40)
TH_FIN Average Grade
Number of Samples
Cross Strike Slice Index

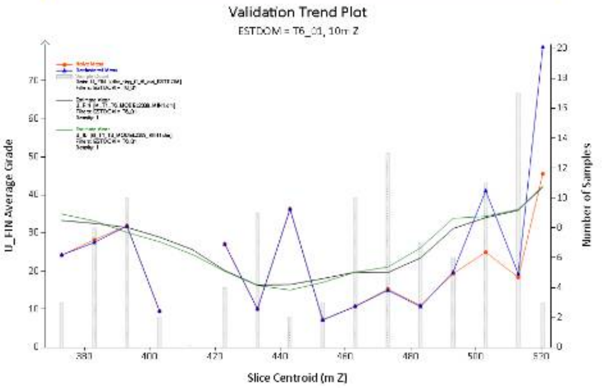

Validation Trend Plot
ESTDOM = T6_01, 10m Z
TH_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
ESTDOM = T6_01, 40m STRIKE(310)
U_FIN Average Grade
Number of Samples
Strike Slice Index

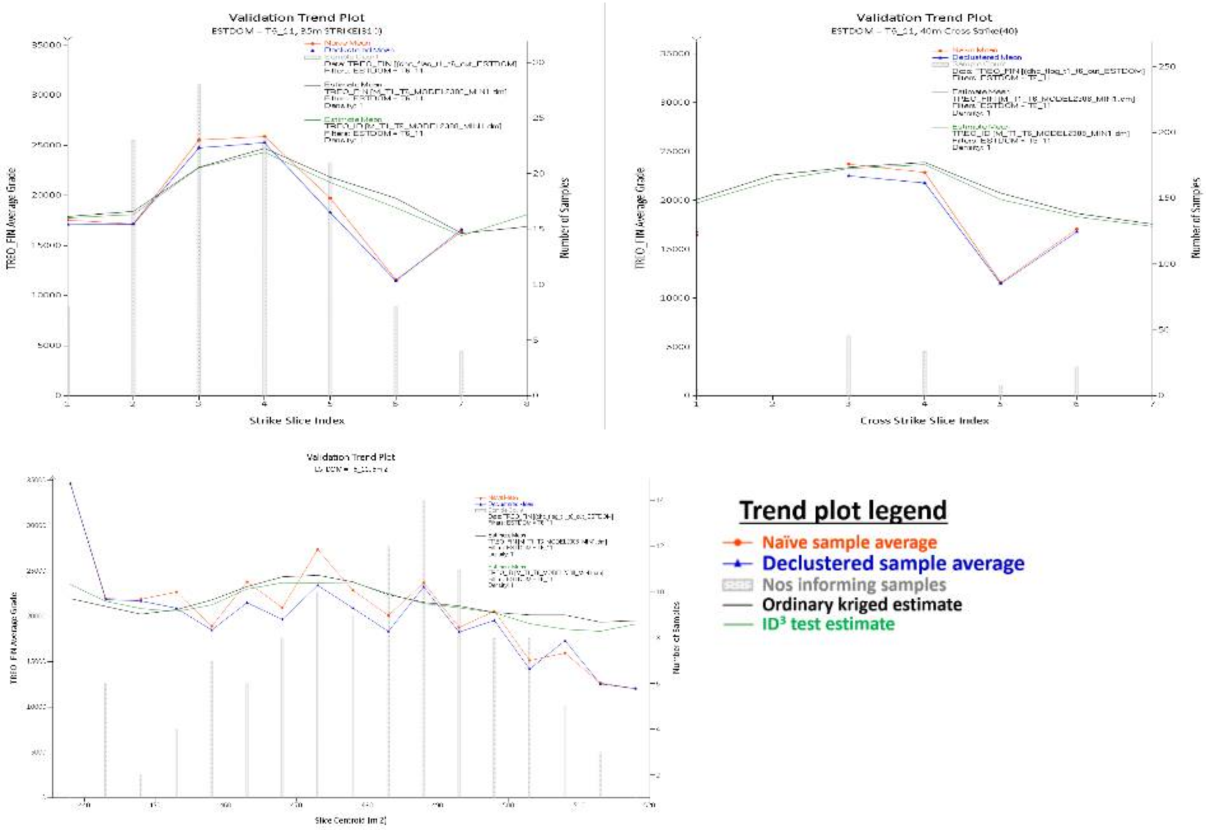

Validation Trend Plot
ESTDOM = T6_01, 40m Cross Strike(40)
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index


Validation Trend Plot
ESTDOM = T6_01, 10m Z
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

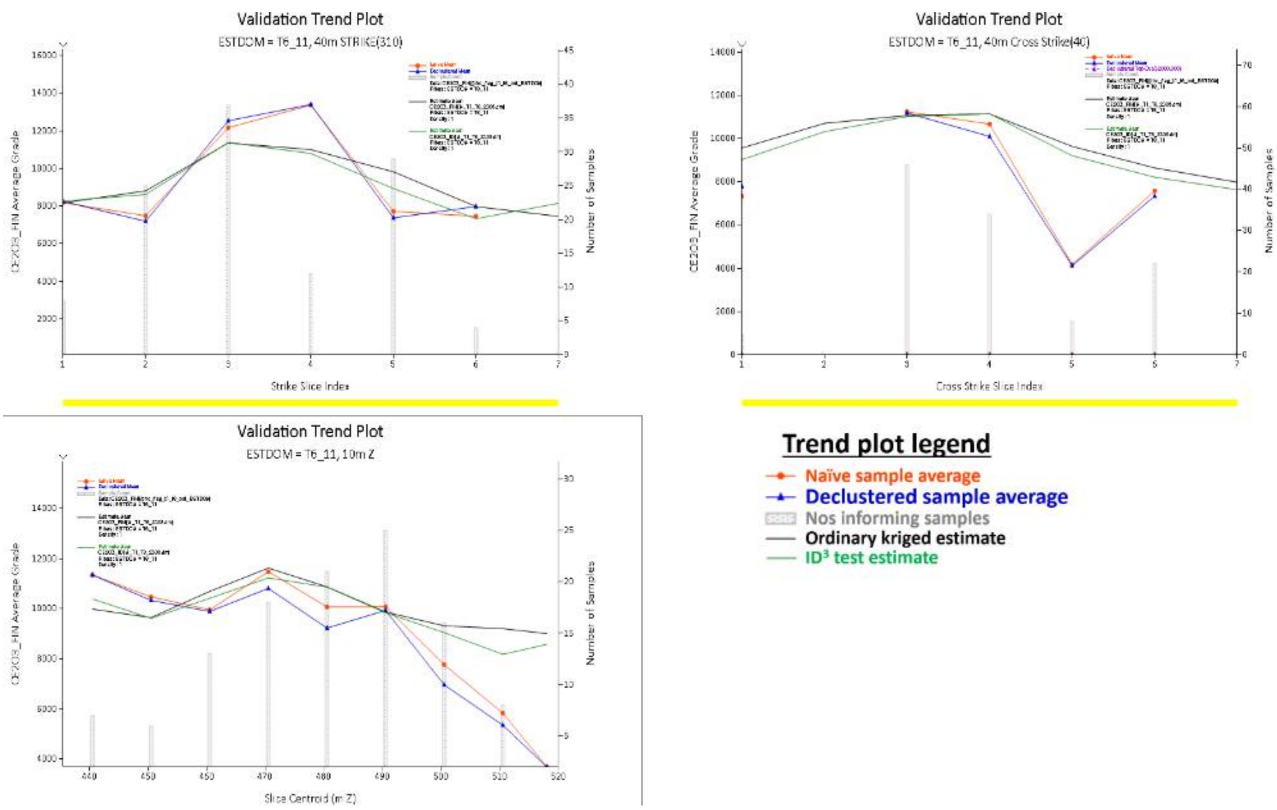

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Model validation trend plots target 6 – T6_11

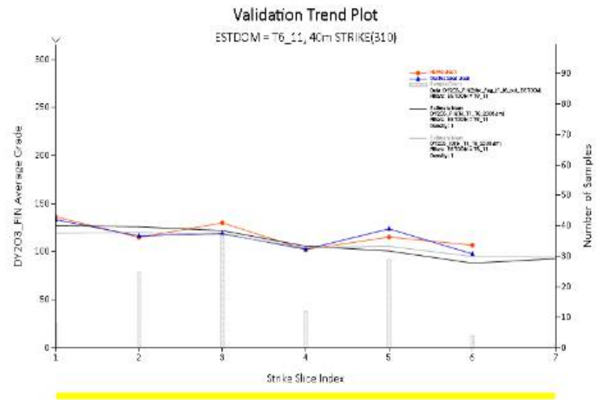
TREO

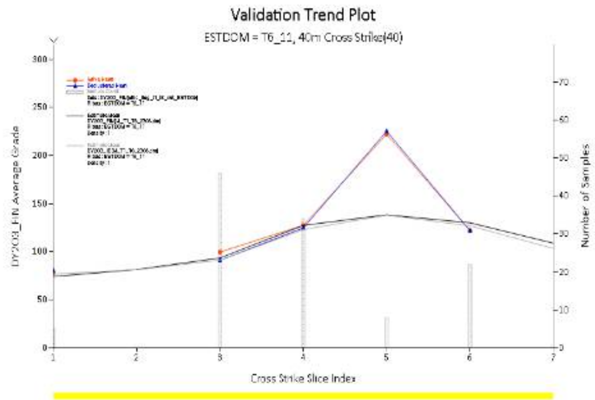


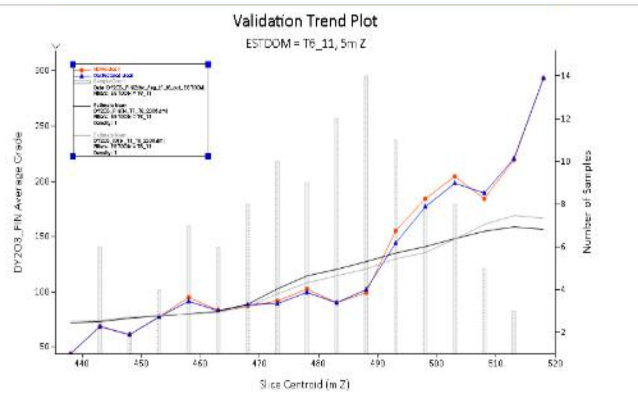
Ce_2O_3

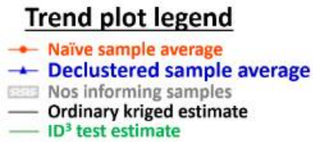


Dy_2O_3

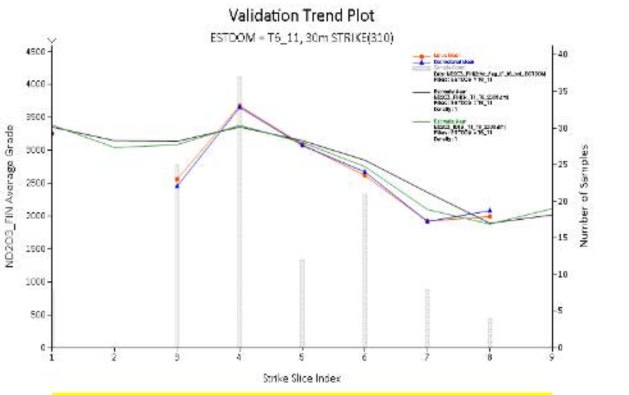

Validation Trend Plot
ESTDOM = T6_11, 40m STRIKE(310)
DY2O3_FIN Average Grade
Number of Samples
Strike Slice Index

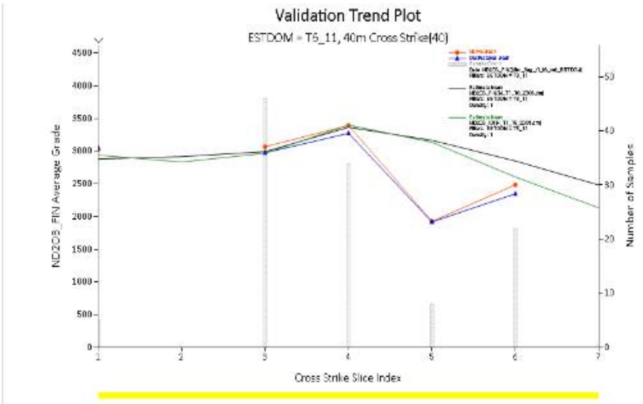

Validation Trend Plot
ESTDOM = T6_11, 40m Cross Strike(40)
DY2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

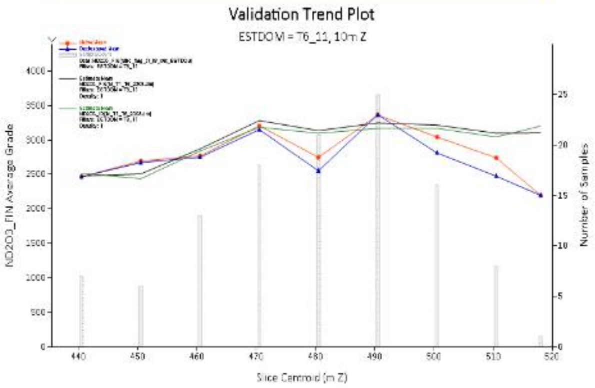

Validation Trend Plot
ESTDOM = T6_11, 5m Z
DY2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

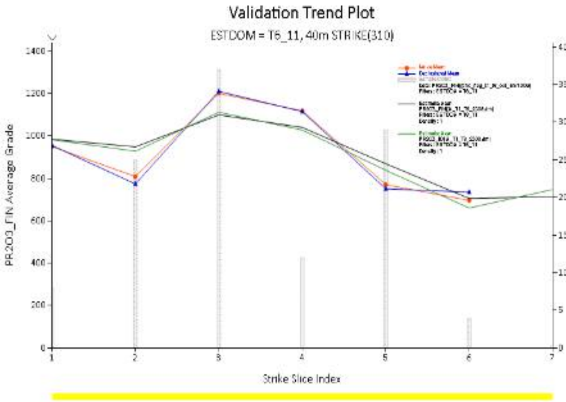

Validation Trend Plot
ESTDOM = T6_11, 30m STRIKE(310)
ND2O3_FIN Average Grade
Number of Samples
Strike Slice Index

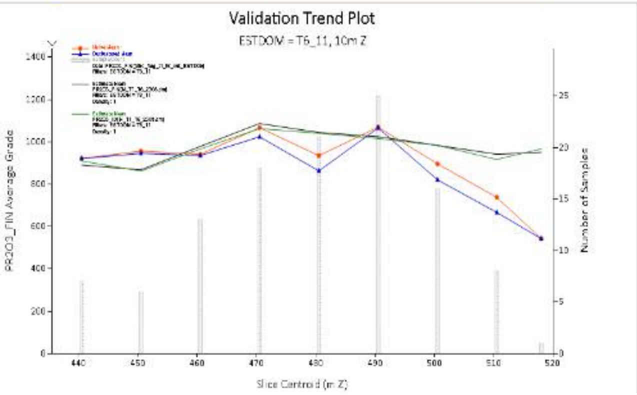

Validation Trend Plot
ESTDOM = T6_11, 40m Cross Strike(40)
ND2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index


Validation Trend Plot
ESTDOM = T6_11, 10m Z
ND2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3

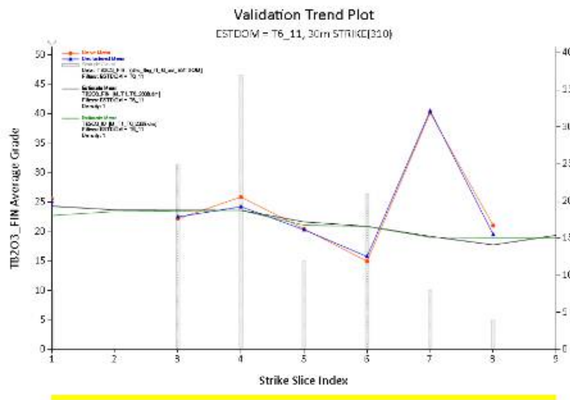


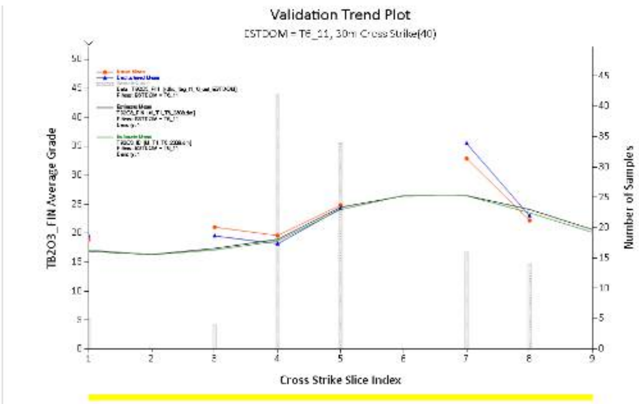


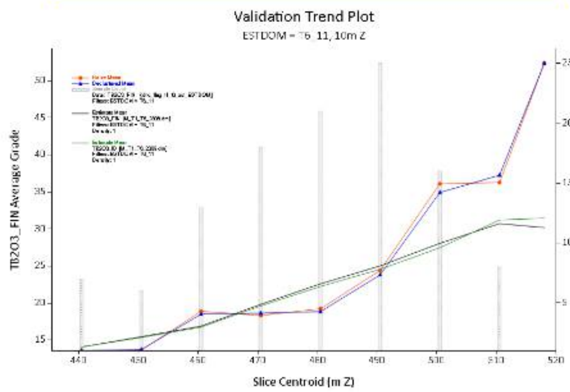
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Tb_2O_3



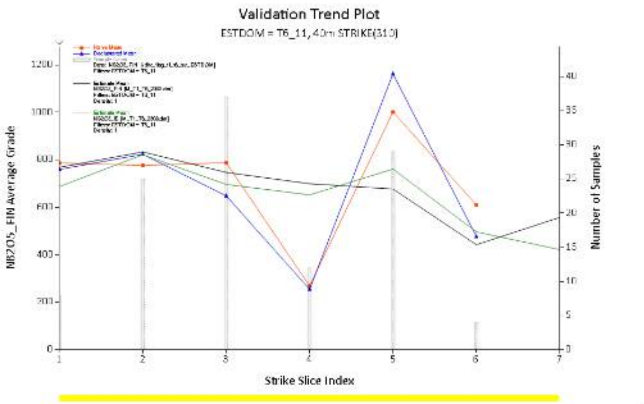


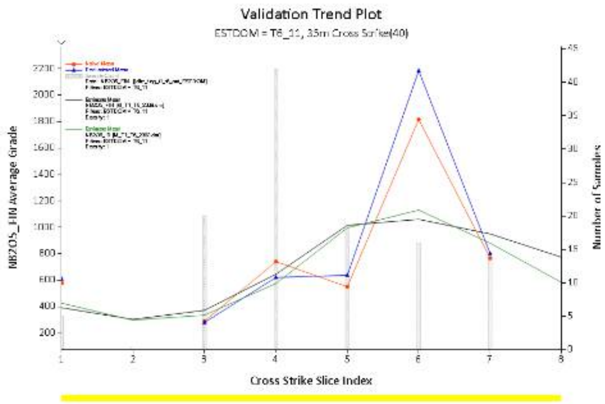


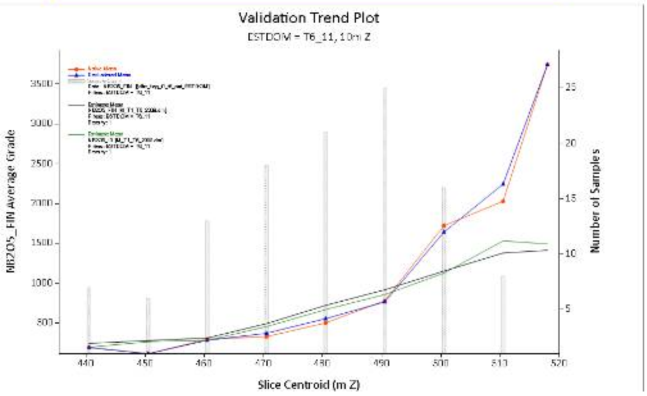
Trend plot legend

- Naïve sample average
- Declustered sample average
- Nos informing samples
- Ordinary kriged estimate
- ID³ test estimate

Nb_2O_5

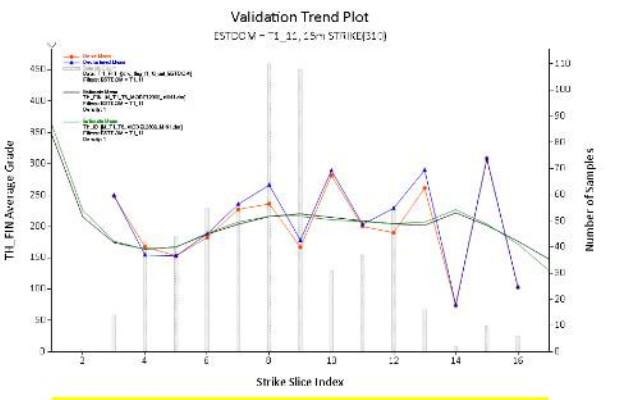

Validation Trend Plot
ESTDOM = T6_11, 40m STRIKE(310)
NB2O5_FIN Average Grade
Number of Samples
Strike Slice Index

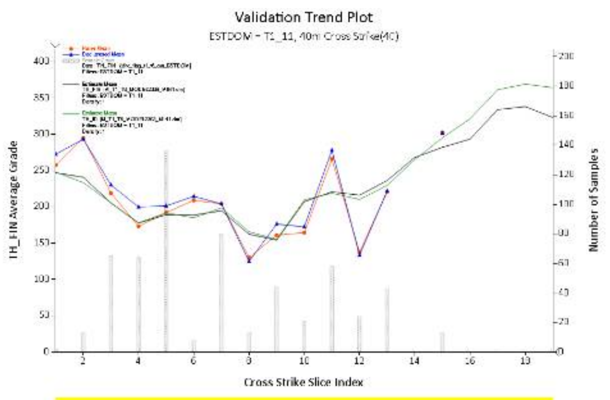

Validation Trend Plot
ESTDOM = T6_11, 35m Cross Strike(40)
NB2O5_FIN Average Grade
Number of Samples
Cross Strike Slice Index

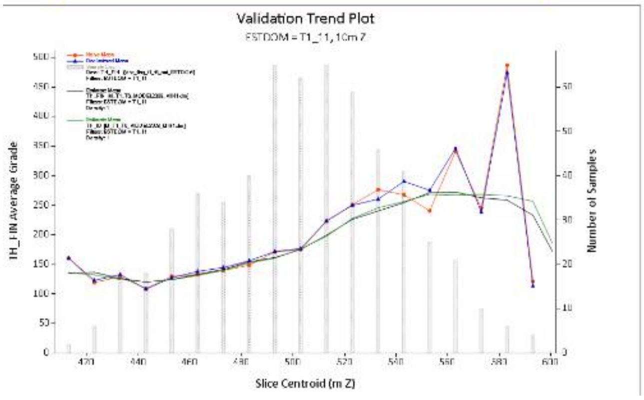

Validation Trend Plot
ESTDOM = T6_11, 10m Z
NB2O5_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

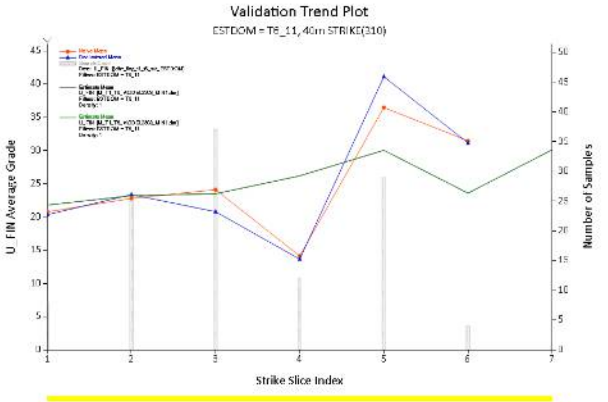

Validation Trend Plot
ESTDOM = T1_11, 15m STRIKE(310)
TH_FIN Average Grade
Number of Samples
Strike Slice Index

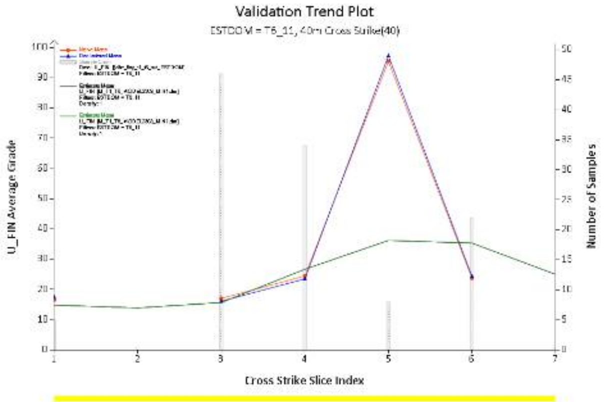

Validation Trend Plot
ESTDOM = T1_11, 40m Cross Strike(40)
TH_FIN Average Grade
Number of Samples
Cross Strike Slice Index

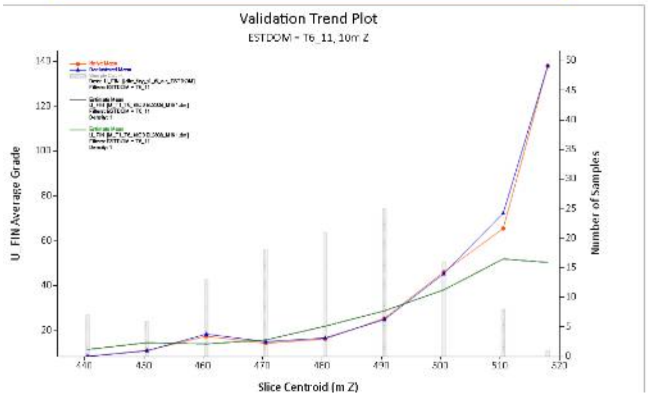

Validation Trend Plot
ESTDOM = T1_11, 10m Z
TH_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
ESTDOM = T6_11, 40m STRIKE(310)
U_FIN Average Grade
Number of Samples
Strike Slice Index

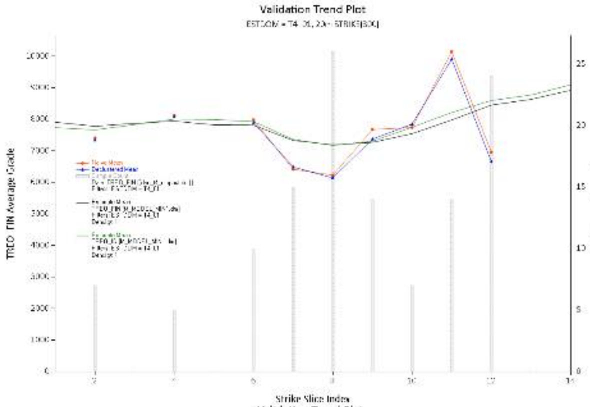

Validation Trend Plot
ESTDOM = T6_11, 40m Cross Strike(40)
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index

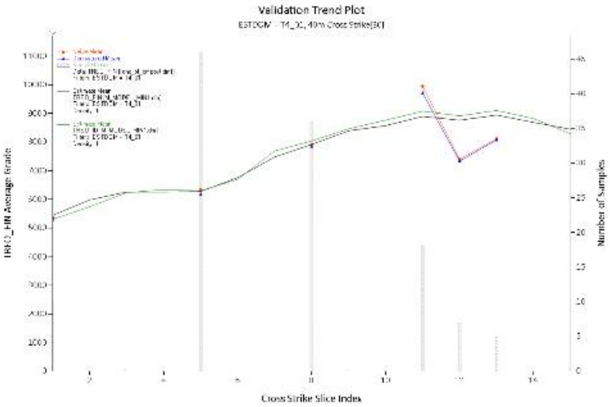

Validation Trend Plot
ESTDOM = T6_11, 10m Z
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

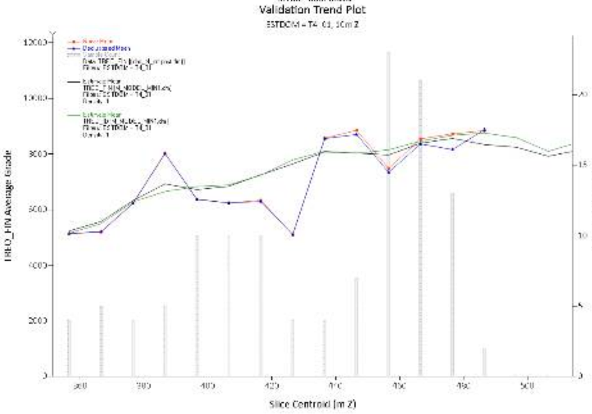

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Model validation trend plots target 4 – T4_01

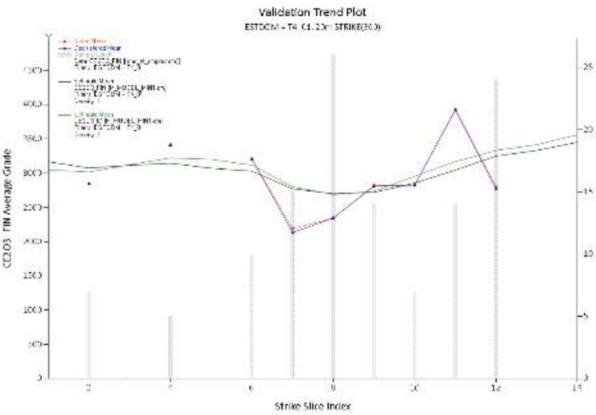
TREO

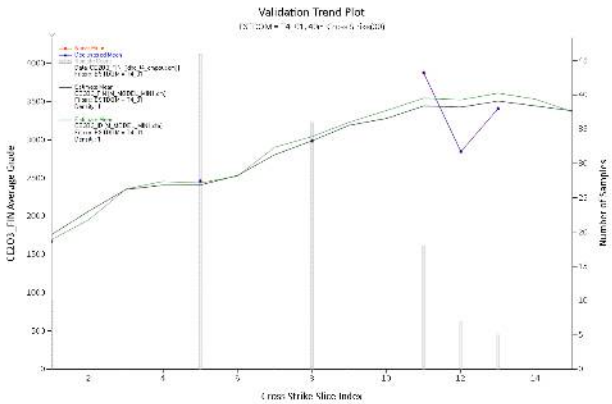


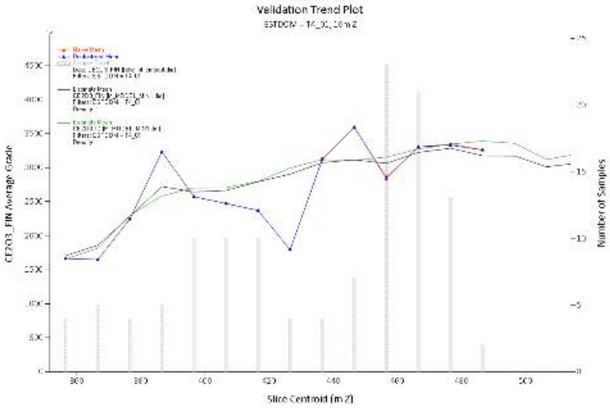




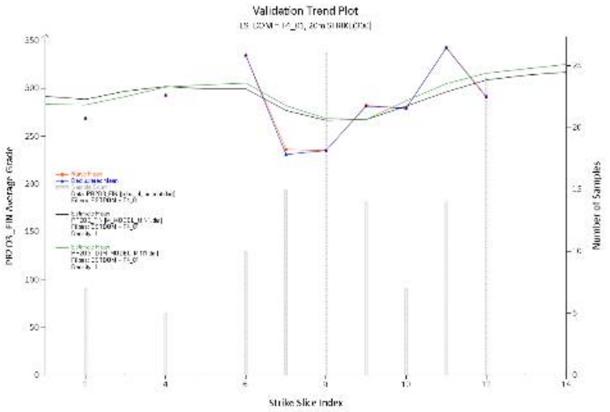
Ce_2O_3

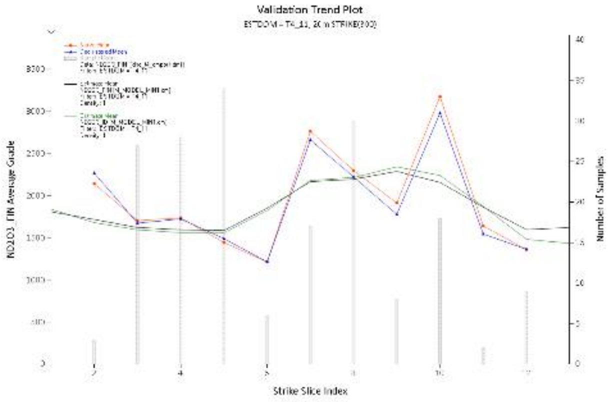


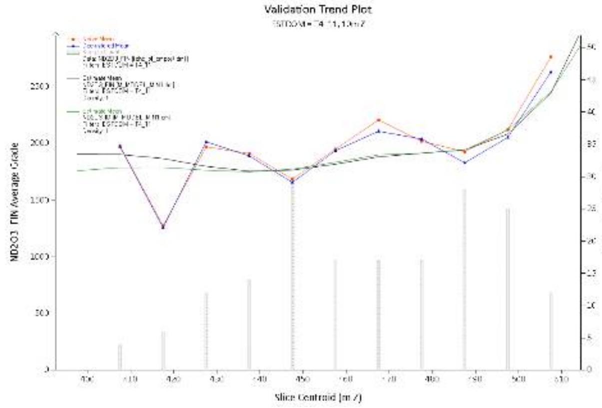


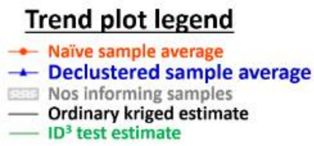


Dy_2O_3

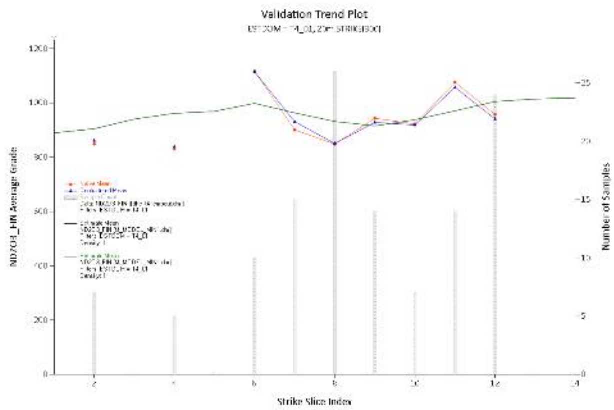

Validation Trend Plot
Strike Slice Index
Number of Samples

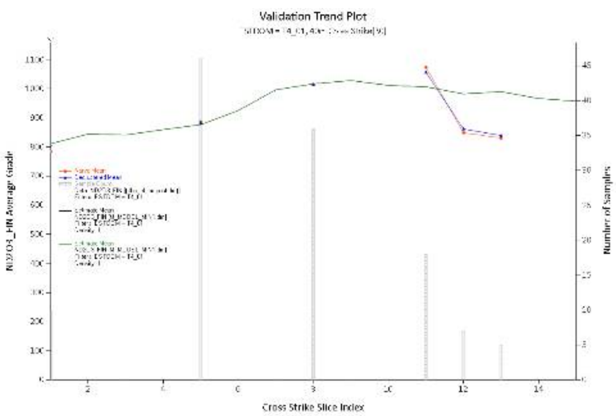

Validation Trend Plot
Strike Slice Index
Number of Samples

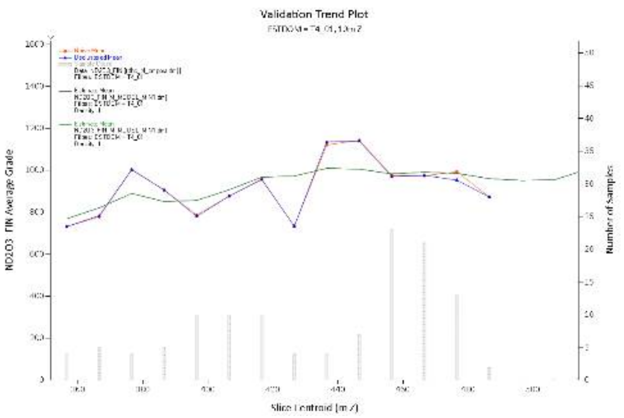

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples

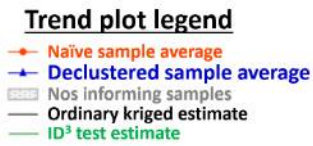

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

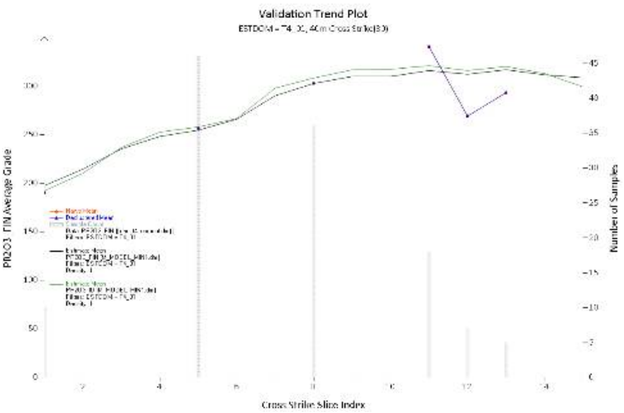

Validation Trend Plot
Strike Slice Index
Number of Samples

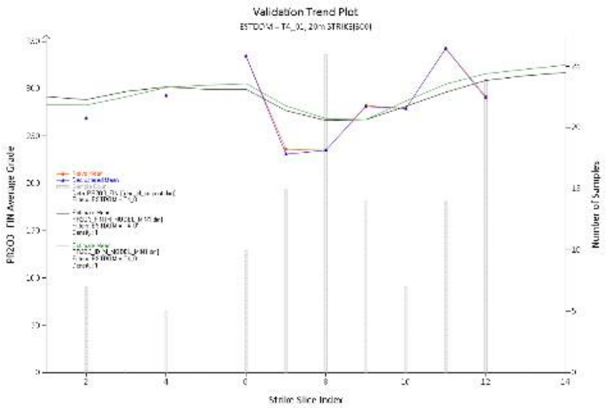

Validation Trend Plot
Cross Strike Slice Index
Number of Samples

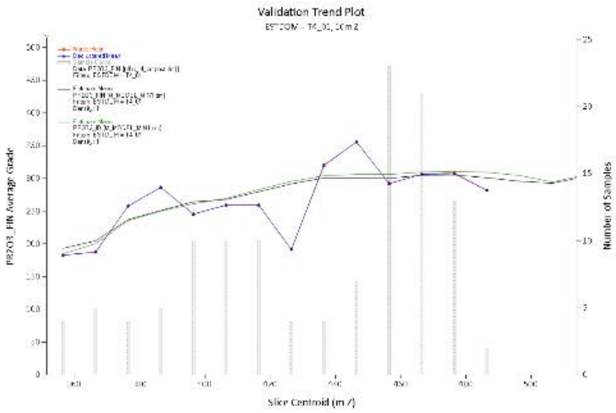

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples

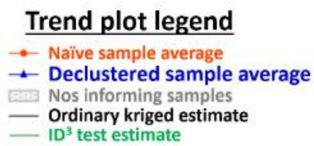

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3

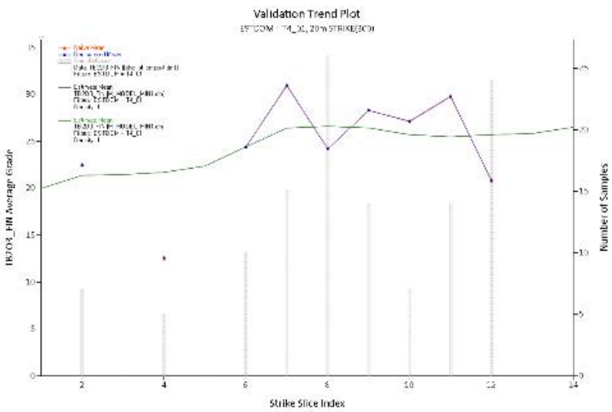

Validation Trend Plot
PR2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

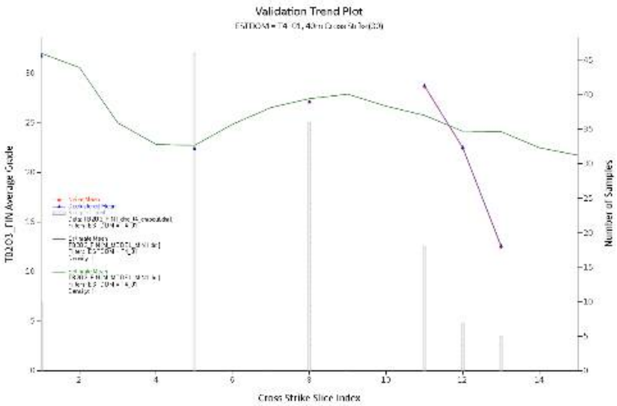

Validation Trend Plot
PR2O3_FIN Average Grade
Number of Samples
Strike Slice Index

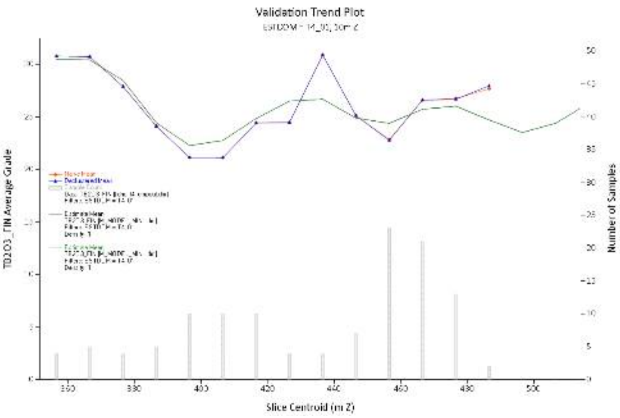

Validation Trend Plot
PR2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Tb_2O_3

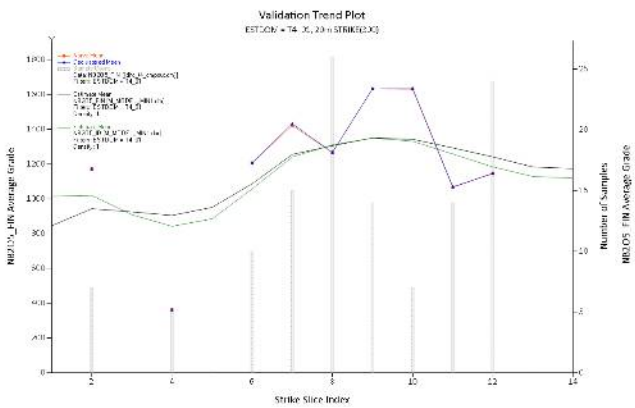

Validation Trend Plot
TB2O3_FIN Average Grade
Number of Samples
Strike Slice Index

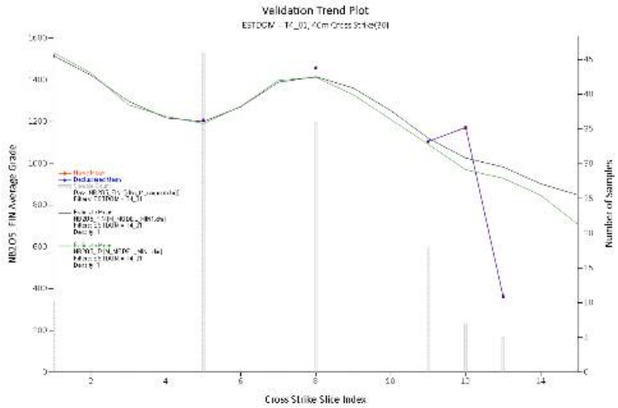

Validation Trend Plot
TB2O3_FIN Average Grade
Number of Samples
Cross Strike Slice Index

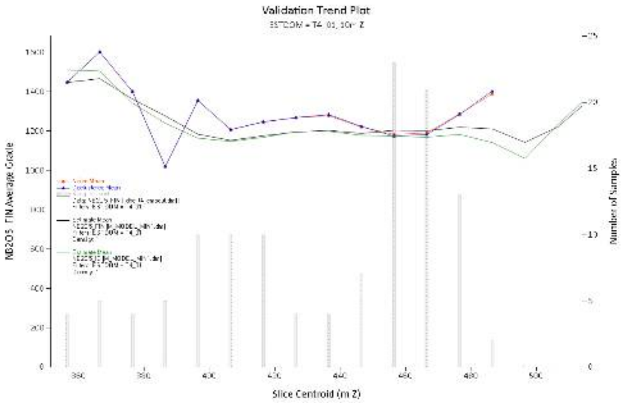

Validation Trend Plot
TB2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nb_2O_5

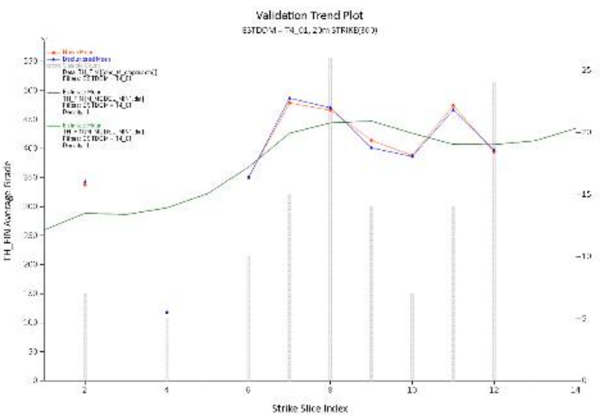

Validation Trend Plot
Strike Slice Index
Number of Samples


Validation Trend Plot
Cross Strike Slice Index
Number of Samples


Validation Trend Plot
Slice Centroid (m Z)
Number of Samples


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th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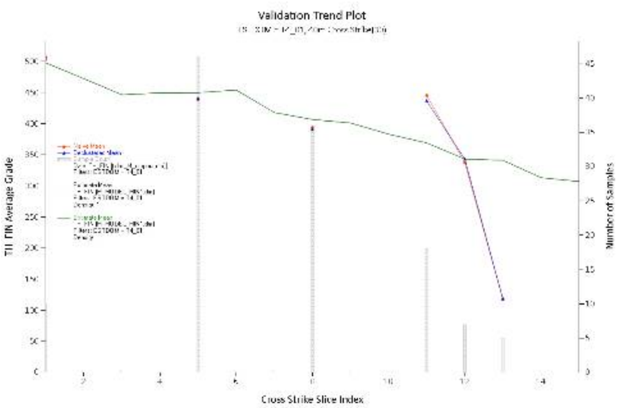
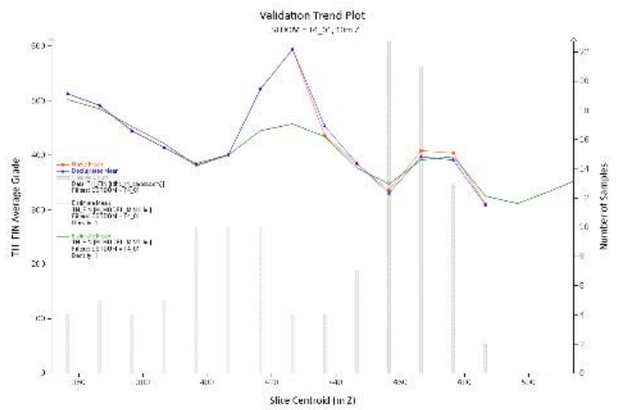
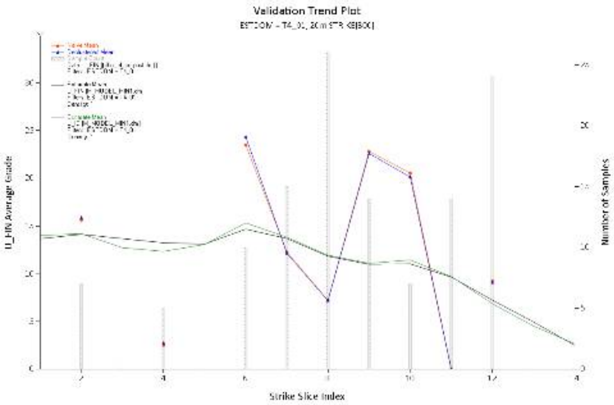
Strike Slice Index
Number of Samples

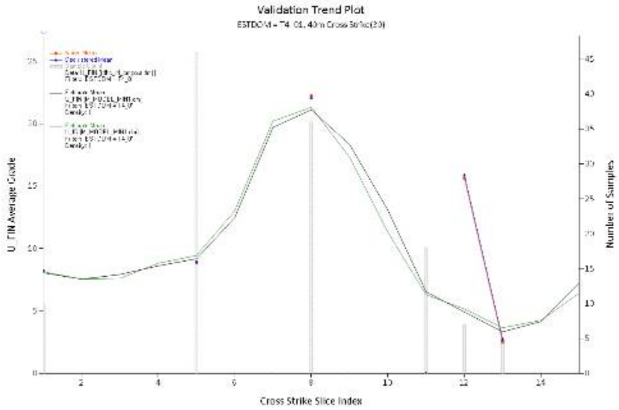

Validation Trend Plot
Cross Strike Slice Index
Number of Samples

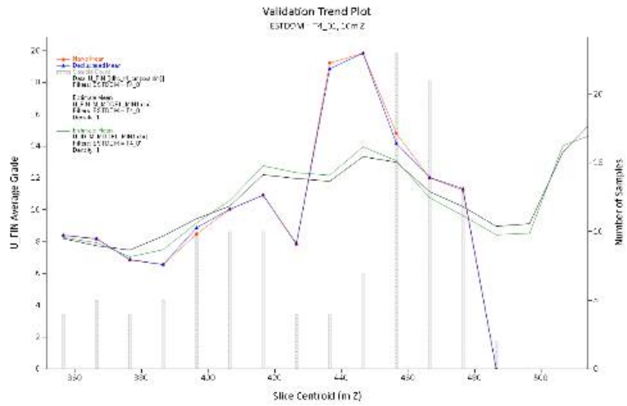

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples

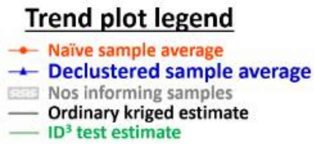

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
U_FIN Average Grade
Number of Samples
Strike Slice Index

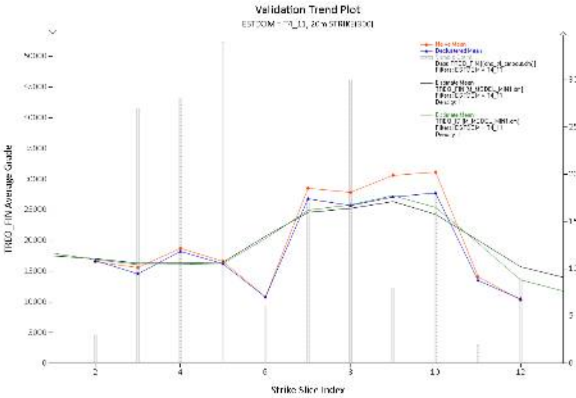

Validation Trend Plot
U_FIN Average Grade
Number of Samples
Cross Strike Slice Index

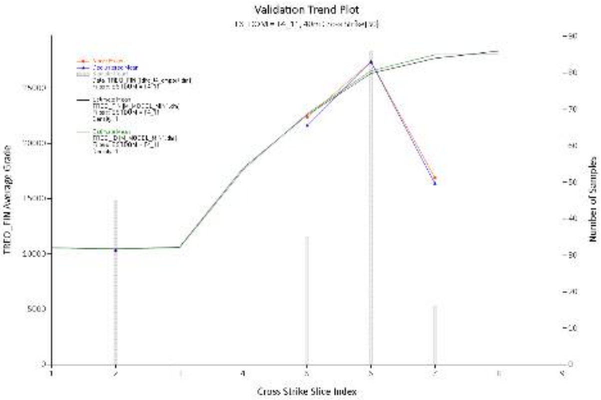

Validation Trend Plot
U_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

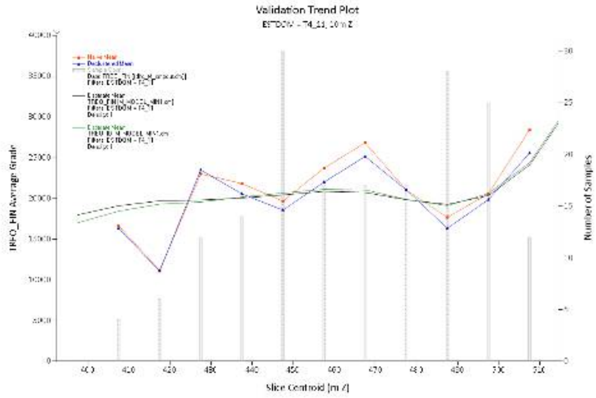

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Model validation trend plots target 4 – T4_11

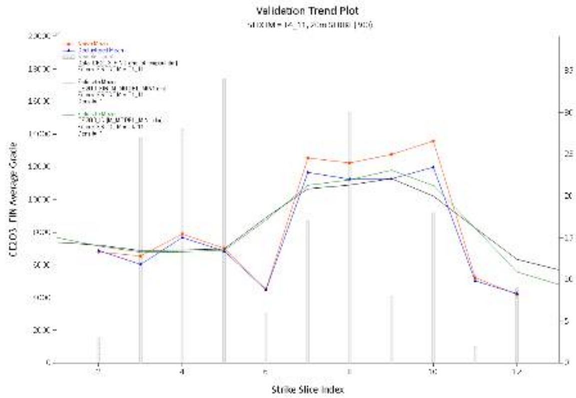
TREO

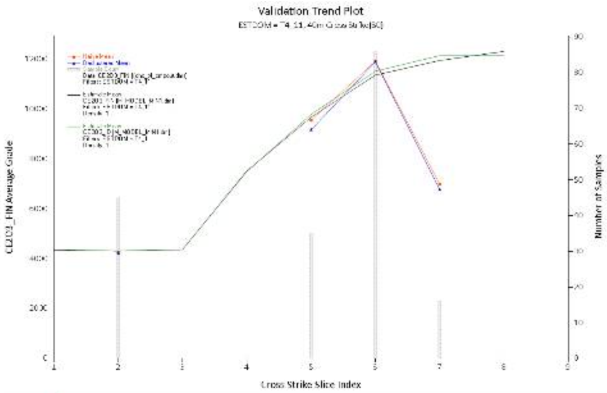


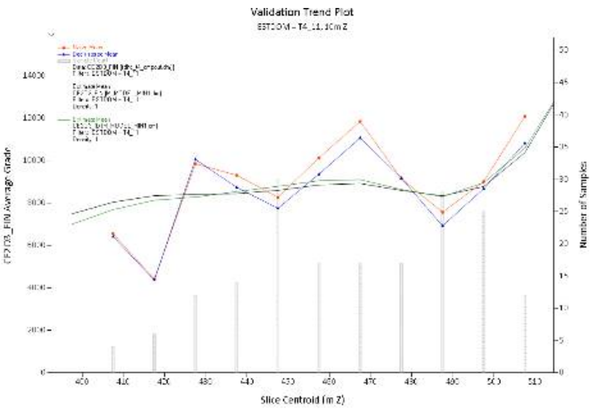




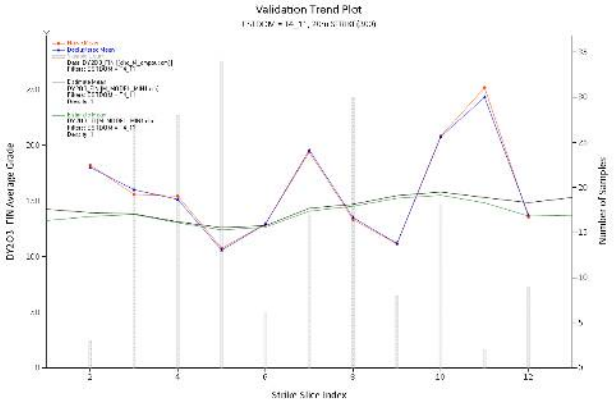
Ce_2O_3

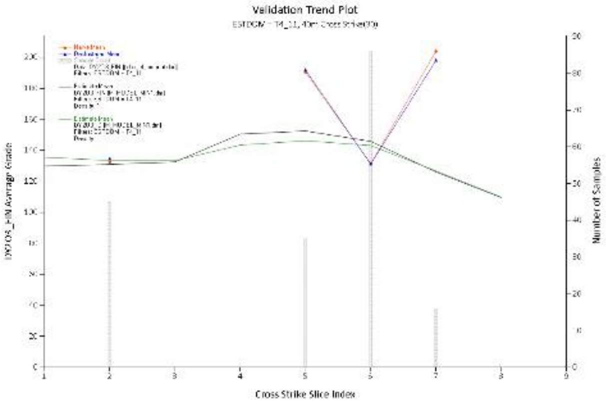


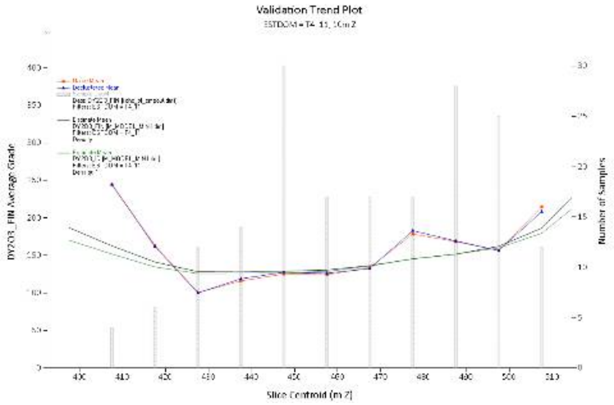


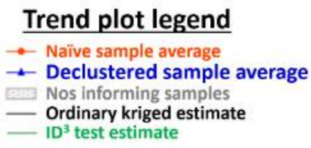


Dy_2O_3

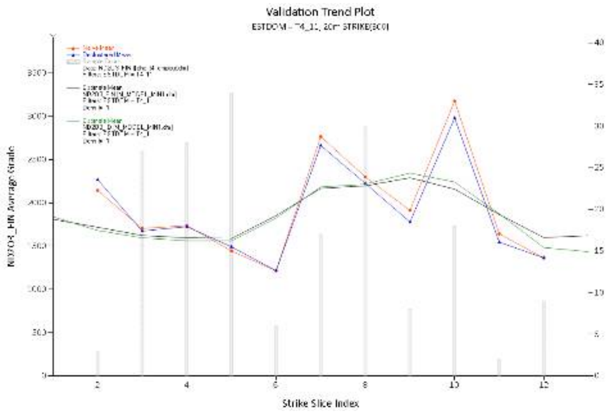

Validation Trend Plot
Strike Slice Index
Number of Samples

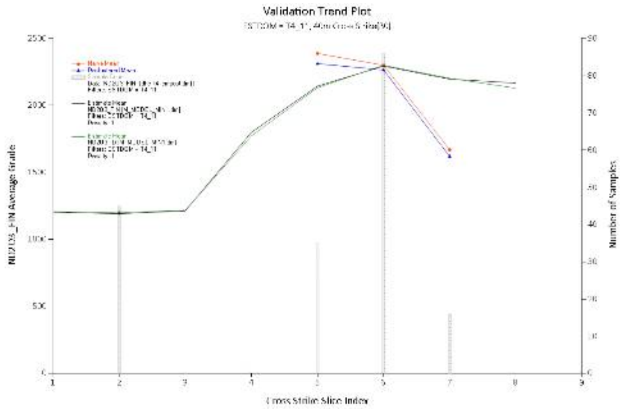

Validation Trend Plot
Cross Strike Slice Index
Number of Samples

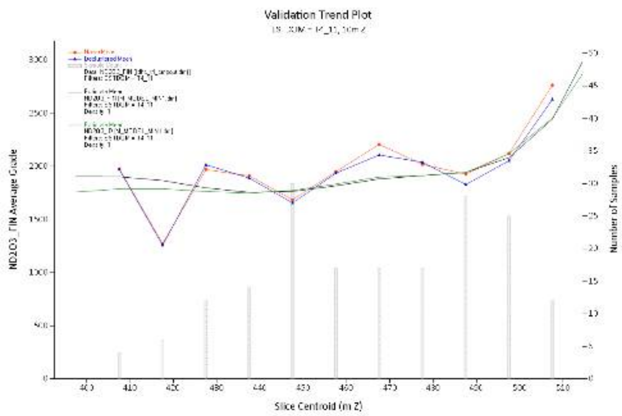

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples

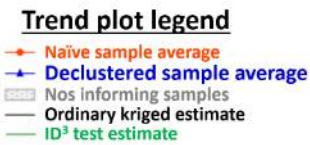

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nd_2O_3

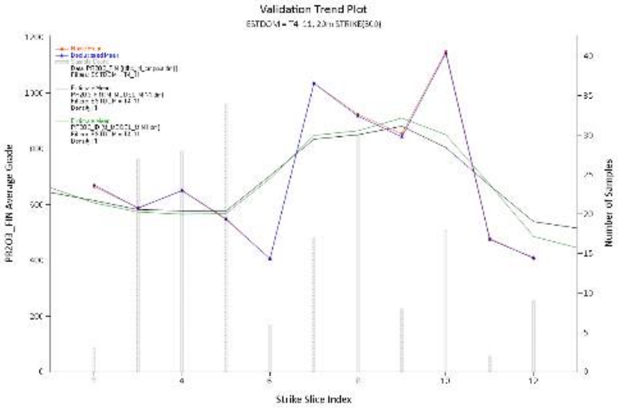

Validation Trend Plot
Strike Slice Index
Number of Samples

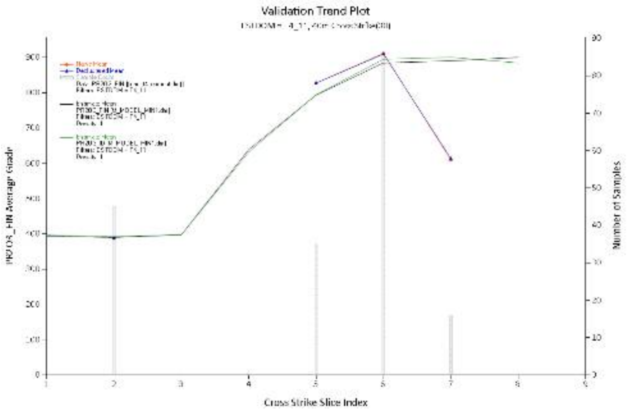

Validation Trend Plot
Cross Strike Slice Index
Number of Samples

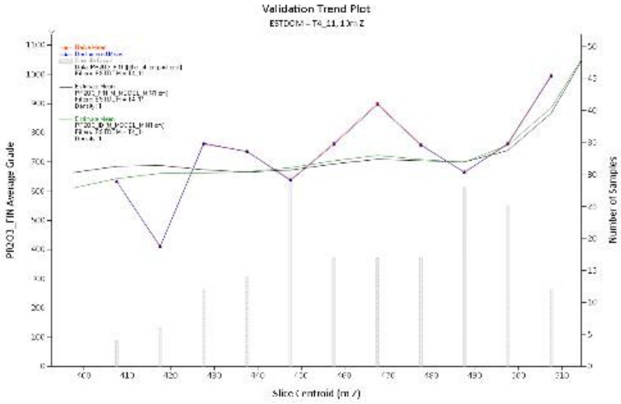

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples

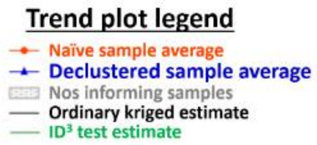

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Pr_2O_3

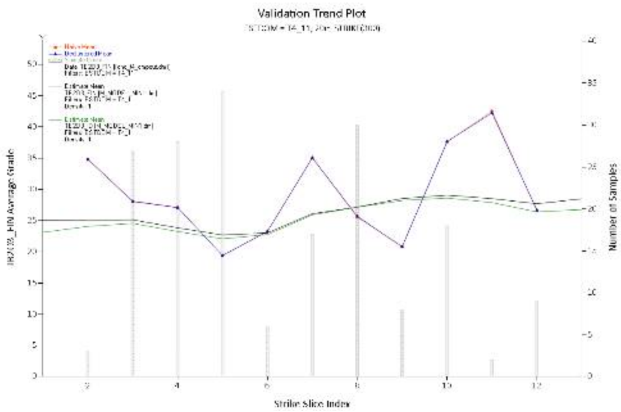

Validation Trend Plot
PR2O3_FIN Average Grade
Number of Samples
Strike Slice Index

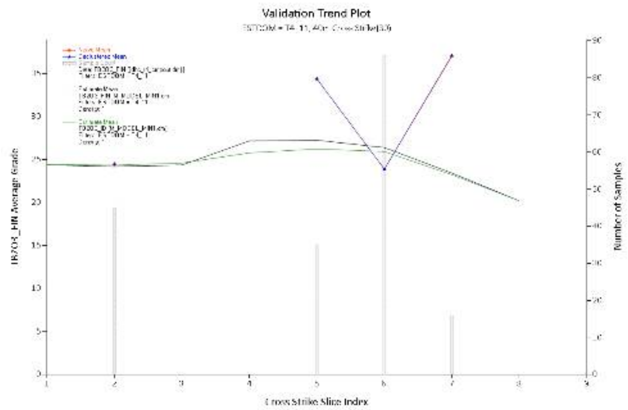

Validation Trend Plot
Number of Samples
Cross Strike Slice Index

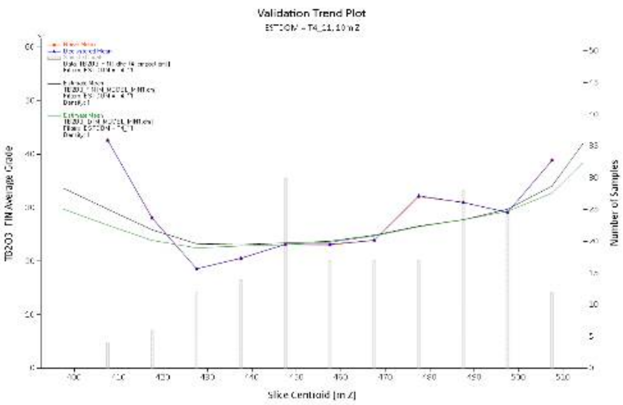

Validation Trend Plot
PR2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

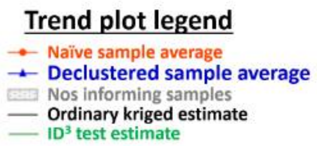

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Tb_2O_3

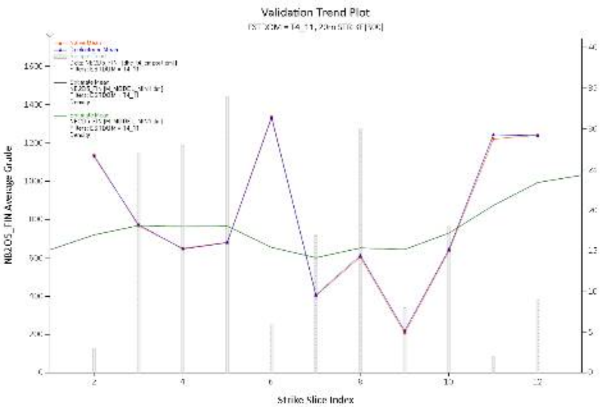

Validation Trend Plot
Number of Samples
Strike Slice Index

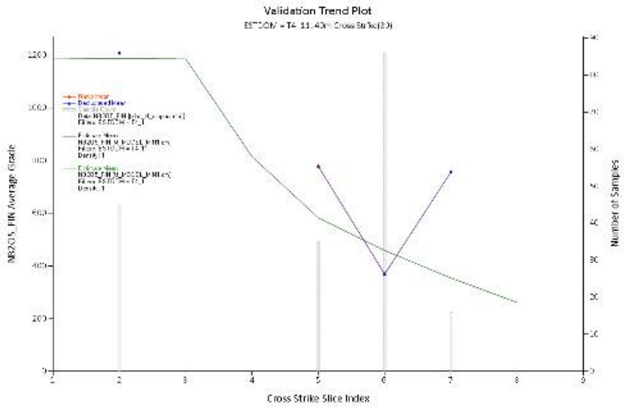

Validation Trend Plot
Number of Samples
Cross Strike Slice Index

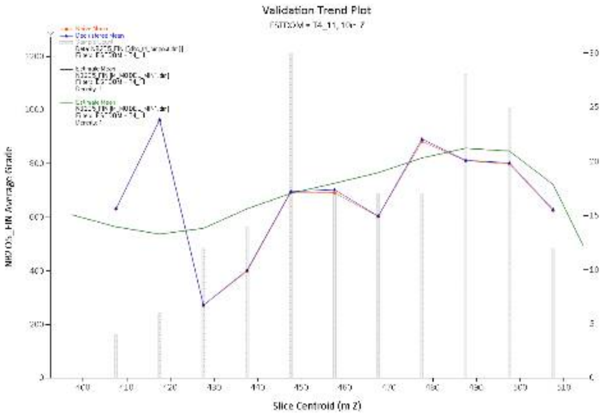

Validation Trend Plot
TB2O3_FIN Average Grade
Number of Samples
Slice Centroid (m Z)

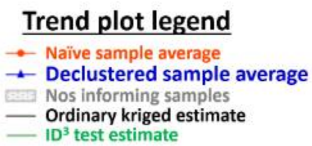

Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Nb_2O_5

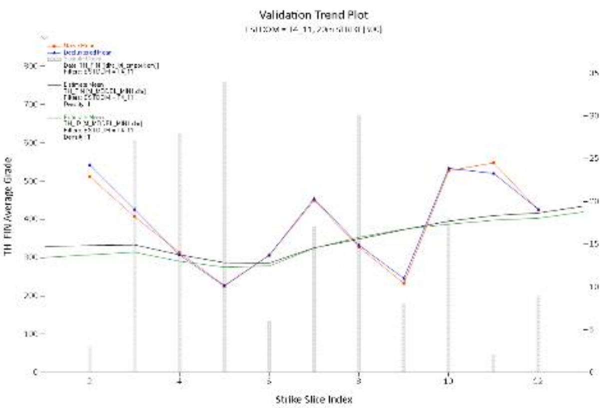

Validation Trend Plot
Strike Slice Index
Number of Samples

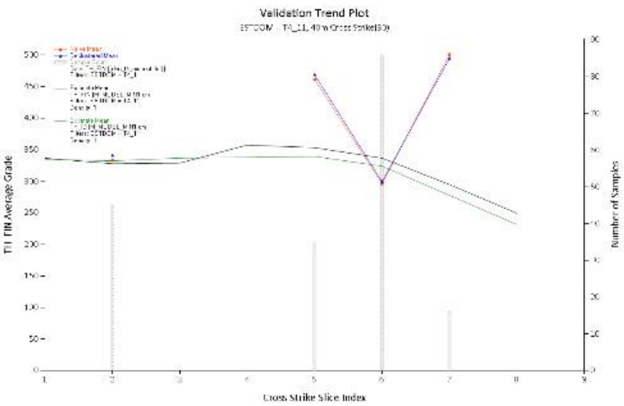

Validation Trend Plot
Cross Strike Slice Index
Number of Samples

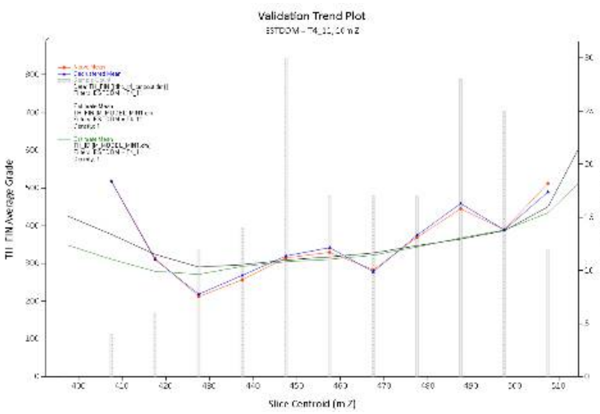

Validation Trend Plot
Slice Centroid (m Z)
Number of Samples


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

Th

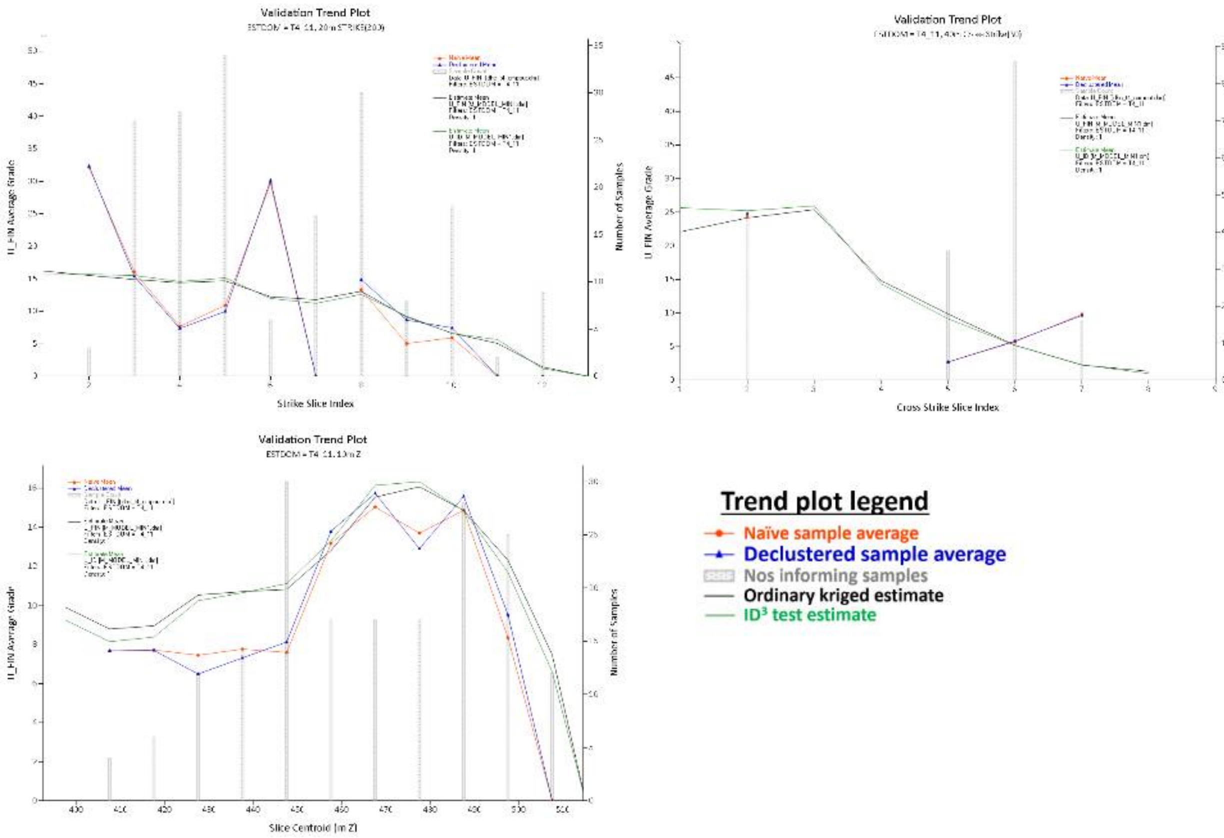

Validation Trend Plot
TH_FIN Average Grade
Strike Slice Index
Number of Samples


Validation Trend Plot
TH_FIN Average Grade
Cross Strike Slice Index
Number of Samples


Validation Trend Plot
TH_FIN Average Grade
Slice Centroid (m Z)
Number of Samples


Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate

U


Validation Trend Plot
U_FIN Average Grade
Number of Samples
Strike Slice Index
Validation Trend Plot
Cross Strike Slice Index
Validation Trend Plot
Slice Centroid (m Z)
Trend plot legend
Naïve sample average
Declustered sample average
Nos informing samples
Ordinary kriged estimate
ID³ test estimate