INDEPENDENT SPECIALIST REPORT

ON THE

AMPASINDAVA RARE EARTHS PROJECT, ANTSIRANANA PROVINCE, MADAGASGAR

WGS84 UTM Zone 38S 191,500 m E; 8,467,900 m N
LATITUDE 13.8421°S, LONGITUDE 48.1459°E

Prepared for:

Harena Resources Pty Ltd
Mount Claremont, Perth, Western Australia 6010, Australia

Technical Assessment Date: August 30th, 2024
MRE Effective Date: November 1st, 2023

Independent Specialist / Competent Person
Yann Camus, P.Eng.
Joseph Keane, P.E.

Company

SGS Geological Services (“SGS”)
SGS Bateman Engineering Services (“SGS”)

SGS Project # 19993-01

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1 EXECUTIVE SUMMARY

SGS Geological Services ("SGS") was contracted by Harena Resources Pty Ltd ("Harena Resources" or the "Company") to complete a Technical Assessment of the Ampasindava Rare Earth project (the "Project" or the "Property") and to prepare an Independent Specialist Report ("ISR" or the "Report"). SGS was also commissioned by Harena Resources to review and restate the MRE in accordance with the guidelines of the JORC 2012 Code in 2023. The ISR is to be included in a prospectus issued by the Company and dated on or about October 2024.

The purpose of this ISR is to provide an independent Technical Assessment of the geology, Mineral Resource estimates ("MRE"), mineral processing, and technical risks associated with the Herena Resources exploration assets, to provide a series of recommendations to advance the Project, and to assess the suitability of the proposed exploration and development programs.

License grant PR 6698 is situated approximately 40 km south-west of Ambanja and covers an area of 237.5 km² in the Antsiranana province of northern Madagascar. The current 100% beneficial owner of exploration licence PR 6698, currently under conversion to an exploitation licence, is Reenova Rare Earth (Malagasy) Sarl.u ("RREM"), a Malagasy company. Harena Resources Pty Ltd have acquired the parent company of RREM to hold a 75% interest in the Property. Access to the Property is by road from Ambanja or by boat from Nosy Be or the mainland port of Ankify.

The Property was first recognised for hosting mineralised dykes hosting high-grade Rare Earth Elements (REE) in fresh rock. REE hosted within the regolith (or soil profile) was recognised in 2012; and it has become clear that this "ionic clay" style of mineralization has the most significant economic potential. The ionic clays are the most important mineralization type for the Project due to the relatively simple process required to put the REE into solution and the fact that neither thorium nor uranium is concentrated through this process; this is in stark contrast to most REE deposits being developed or promoted in the western world. Additionally, the geological setting of the Project is analogous to the source of most of the Heavy REE (HREE) currently produced in China.

A massive amount of data has been collected by excavating shafts by hand and sampled along the depth. A total of 4,474 test pits have been completed with an average depth of 5.8m with a maximum depth of 10m. The sampling methodology was validated in the field by SGS Geostat and this has produced reliable data that has been integrated into the following resource estimation. The limitations of this method include safety considerations (including a 10m depth limit) and the impossibility of collecting data below the water table. A total of 4,412 pits were retained for the resource estimation along with 359 drill holes.

The updated resource estimation was constructed solely from the laterite and saprolite layers within the Property. REE grades were interpolated separately by ordinary kriging and blocks were cut-off based on their conceptual amenability to heap leaching. The base case resource with tonnage and grade is presented in Table 1-1.

Table 1-1 Mineral Resource Estimate for Ampasindava Deposit at Cut-Off of 500 ppm TREO

Classification	Tonnage (Mt)	Volume (Mm³)	Area (Mm²)	Density (t/m³)	Thickness (m)			TREO (ppm)	MREO (ppm)	MREO / TREO ratio	Contained TREO (t)	Contained MREO (t)
					Total	PED	SAP					
Measured	42.5	38.1	7.0	1.11	5.46	2.85	2.60	958	221	23%	40,700	9,400
Indicated	184.0	167.1	25.0	1.10	6.70	2.65	4.04	842	178	21%	154,800	32,700
Measured + Indicated	**226.5**	**205.3**	**31.9**	**1.10**	**6.43**	**2.70**	**3.73**	**863**	**186**	**22%**	**195,500**	**42,100**
Inferred	472.0	429.1	78.9	1.10	5.44	2.71	2.73	870	189	22%	410,500	89,000
Total	698.5	634.3	110.8	1.10	5.72	2.71	3.02	868	188	22%	606,000	131,100

1. *The Mineral Resource Estimate (MRE) has an effective date of the 1st November, 2023. The Competent Person for the MRE is Mr. Yann Camus, P.Eng., an employee of SGS.*
2. *The classification of the current Mineral Resource Estimate is consistent with the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code).*
3. *All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.*
4. *All Resources are presented undiluted and in situ, constrained within a 3D model, and are considered to have reasonable prospects for eventual economic extraction.*
5. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
6. *Bulk density values were determined based on physical test work from each part of the deposit.*
7. *The base cut-off grade of 500ppm TREO considers a mining cost of US$1.40/t mined, a processing cost of $8.00/t mined and G&A cost of US$0.75/t mined.*
8. $TREO = Y_2O_3+Eu_2O_3+Gd_2O_3+Tb_2O_3+Dy_2O_3+Ho_2O_3+Er_2O_3+Tm_2O_3+Yb_2O_3+Lu_2O_3+La_2O_3+Ce_2O_3+Pr_2O_3+Nd_2O_3+Sm_2O_3$
9. $MREO = Pr_2O_3+Nd_2O_3+Tb_2O_3+Dy_2O_3$
10. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

In the 2023 Critical Materials Report from the U.S. Department of Energy, it was reported that three of the 17 rare earth elements were found to be critical in terms of supply risk in the short term (2020 to 2025), with four rare earth elements identified as critical into the medium term (2025 – 2035) (Figure 4-7). These elements are termed “critical” because of their scarcity. This, combined with anticipated growth in demand, makes these higher valued elements the ones expected to experience the best price performance over the next decade. Those elements deemed critical over the next decade are: dysprosium, neodymium, praseodymium, and terbium. These four rare earth elements (MREO) are used in magnets for wind turbines and electric vehicles, as phosphors in energy-efficient lighting, and also in high-tech strategic applications.

The Ampasindava Ionic clay project displays a pervasive and well-balanced rare earth distribution that is prevalent throughout the deposit. The deposit includes appreciable amounts of the critical magnet rare earths dysprosium, neodymium, praseodymium, and terbium as defined by the U.S. Department of Energy.

The Ampasindava Project is one of relatively few projects of ionic clay hosted REE deposits outside of China. This sets it apart from most of the developers vying to be a stable source of these metals (particularly the HREE) for the western world.

Work to date on the Project has focused on outlining resources; this goal has largely been completed with a significant mineral inventory that can be incorporated in economic studies, and on metallurgical test work to establish the amenability of the deposit to leach processing methodologies. Risks and recommendations are presented in Section 6 of this report. Many studies should be conducted in parallel at this stage to mitigate the various risks known for this Project. These studies should culminate in a Feasibility Study which will provide further detail on the economic potential for this Project.

2 INTRODUCTION

2.1 Background

SGS Geological Services (“SGS”) was contracted by Harena Resources Pty Ltd ("Harena Resources" or the "Company") to complete a Technical Assessment of the Ampasindava Rare Earth project (the “Project” or the “Property”) and to prepare an Independent Specialist Report ("ISR" or the “Report”). SGS was also commissioned by Harena Resources to review and restate the MRE in accordance with the guidelines of the JORC 2012 Code in 2023. The ISR is to be included in a prospectus issued by the Company and dated on or about October 2024.

The funds raised will be primarily used for the advancement of economic, social and environmental studies and planning for trial mining and processing of rare earth elements on the Ampasindava project. This ISR details the principal project area surrounding the Ampasindava Rare Earths deposit in Antsiranana Province, Madagascar.

2.2 Scope

The purpose of this ISR is to provide an independent Technical Assessment of the geology, Mineral Resource estimates (“MRE”), mineral processing, and technical risks associated with the Herena Resources exploration assets, to provide a series of recommendations to advance the Project, and to assess the suitability of the proposed exploration and development programs.

This report presents the following key technical information on the date of this Report:

- An overview of the geological setting of the mineral assets and the associated mineralisation;
- Outline of the historical and recent exploration work undertaken;
- Exploration results reported in accordance with the terms and definitions of the JORC Code (2012);
- Mineral Resources reported in accordance with the terms and definitions of the JORC Code (2012);
- Mineral processing and metallurgical test work reported in accordance with the terms and definitions of the VALMIN Code (2015);
- Independent Specialist opinion on the reasonableness of the stated Mineral Resource estimates in light of potential project opportunities and constraints;
- Independent Specialist opinion on the reasonableness of the stated mineral processing test work in light of potential project opportunities and constraints;
- Independent Specialist opinion on the key risks and opportunities related to the exploration and development potential of the project;
- Independent Specialist opinion on the appropriateness of the proposed work programs.

Should SGS determine that an assumption included in the Technical Assessment is unreasonable then this will be reflected in the Report.

2.3 Reporting Standard

This Report has been prepared as a public document to the standard of, and is considered by SGS to be, a Technical Assessment in the format of an Independent Specialist's Report in accordance with the guidelines of the Australasian Code for Public Reporting of Technical Assessments and Valuations of Mineral Assets – the 2015 VALMIN Code (“VALMIN”) and the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves – the 2012 JORC Code (“JORC”).

As defined in the VALMIN Code (2015), Mineral Assets comprise all property including (but not limited to) tangible property, intellectual property, mining and exploration tenure, and other rights held or acquired in connection with the exploration, development of and production from those tenures. This may include the plant, equipment, and infrastructure owned or acquired for the development, extraction, and processing of minerals in connection with that tenure.

A first draft of this report was supplied to Harena Resources to check for material errors, factual accuracy, and omissions before the final report was issued. SGS's Report does not comment on the 'fairness and reasonableness' of any transactions between Harena Resources and any other parties.

2.4 Data Sources

SGS has based its review of the projects on the information made available to the authors by Harena Resources, along with technical reports prepared by consultants, government agencies, previous tenements holders, and other relevant published and unpublished data. SGS has also relied upon discussions with Harena Resource's management for the information contained within this technical report. This Report has been based upon information available up to and including the stated Effective Date.

Much of the information summarised in this Report has been sourced from the previous NI 43-101 Technical Report completed by SGS (Desharnais et al. 2016).

SGS has endeavored, by making all reasonable enquiries, to confirm the authenticity, accuracy, and completeness of the technical data upon which this Report is based. Unless otherwise stated, information and data contained in this technical report or used in its preparation have been provided by Harena Resources in the form of documentation.

Descriptions of the mineral tenure, tenure agreements, encumbrances, and environmental liabilities were provided to SGS by Harena Resources or its technical consultants. Harena Resources has warranted to SGS that the information provided for the preparation of this Report correctly represents all material information relevant to the Project. Full details on the tenements are set out in the Solicitor's Report on Tenements in the Prospectus.

2.5 Site Visit

SGS did not consider that a current site visit was warranted as it was considered that a site visit would not reveal information or data material to the outcome of this Report as there has not been material exploration drilling or on-ground development completed on the Project subsequent to the 2013 site visit completed by Guy Desharnais for SGS and the 2014 NI 43-101 Technical Report completed by SGS. The Independent Specialists are satisfied that there is sufficient current information available to allow an informed evaluation to be made without an inspection.

2.6 Tenement Status Verification

Verification of information concerning Property status and ownership, which are presented in Section 4.5 below, has been provided to SGS by Allan Mulligan of Harena Resources in the form of a legal opinion obtained on February 2nd 2023 from Madagascar Conseil International. SGS only reviewed the land tenure in a preliminary fashion and has not independently verified the legal status or ownership of the Property or any underlying agreements or obligations attached to ownership of the Property. However, SGS has no reason to doubt that the title situation is other than what is presented in this technical report. SGS is not qualified to express any legal opinion with respect to Property titles or current ownership.

Details of the legal ownership of the mineral assets are dealt with in the Solicitor's Report in the Prospectus.

2.7 Effective Dates

The Technical Assessment Date of this Report is August 30th, 2024.
The Effective Date of the current MRE, as detailed in this Report, is November 1st, 2023.

2.8 Project Team

This Report has been prepared by a team of consultant's from SGS's offices in Canada and the United States of America. Details of the qualifications and experience of the consultants who have carried out the work in the Report, who have extensive experience in the mining industry and are members in good standing of appropriate professional institutions are set out in Table 2-1.

Table 2-1 Independent Specialist Qualifications and Experience

Specialist	Position/ Company	Responsibility	Length and type of experience	Site inspection	Professional designation
Yann Camus	Senior Engineer/ SGS	Mineral Resources and Geology	23 years in Mineral Resource estimation, open pit and underground production and reconciliation, and project evaluation	None	P.Eng. registered with the Ordre des Ingenieurs du Quebec—OIQ
Joseph Keane	Principal Process Engineer/ SGS	Metallurgical Testing and Processing	61 years –41years in consulting specialising in engineering design, metallurgical laboratory management, and technical reviews; 20 years in operations	None	P.E. Society for Mining, Metallurgy, and Exploration, Inc. (SME license No. 1682600) Registered Professional Metallurgical Engineer in Arizona (License No. 12979) Registered Professional Metallurgical Engineer in Nevada (License No. 5462)

2.9 Independence

The Independent Specialists have no material present or contingent interest in the mineral assets reviewed. Neither SGS nor the authors of this Report have or had previously any material interest in Harena Resources or the mineral properties in which Harena Resources has an interest.

For clarity, two of the authors of this Report have previously reviewed these mineral assets and completed Technical Assessments to estimate Mineral Resources and hence have a good understanding or the Project geology and Mineral Resources. SGS has previously completed metallurgical testing on the Project in the period from 2013 to 2014 to evaluate appropriate mineral processing systems for the Project.

This Report was commissioned by Harena Resources on a fee-for-service basis according to SGS' schedule of rates depending on the consultant's skills and experience. SGS' fee is not contingent on the outcome of the initial public offer to be conducted by Harena Resources.

2.10 Disclaimer and Indemnities

SGS's opinion contained herein is based on information provided to SGS by Harena Resources throughout the course of SGS's investigations as described in this Report, which in turn reflect various technical and economic conditions at the time of writing. Such technical information as provided by Harena Resources was taken in good faith by SGS. SGS has recently recalculated the Mineral Resources (October 2023) and has independently assessed the reasonableness of the estimates.

This Report includes technical information, which requires subsequent calculations to derive subtotals, totals, averages, and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur, SGS does not consider them to be material.

As far as SGS has been able to ascertain, the information provided by Harena Resources was complete and not incorrect, misleading, or irrelevant in any material aspect.

Harena Resources has confirmed to SGS in writing that full disclosure has been made of all material information and that to the best of its knowledge and understanding, the information provided by Harena Resources was complete, accurate, and true and not incorrect, misleading, or irrelevant in any material aspect. SGS has no reason to believe that any material facts have been withheld.

As recommended in by the VALMIN Code (2015), Harena Resources has provided SGS with an indemnity under which SGS is to be compensated for any liability and/or any additional work or expenditure, which:

- results from SGS's reliance on information provided by Harena Resources and/or independent consultants that are materially inaccurate or incomplete; or
- relates to any consequential extension of workload through queries, questions or public hearings arising from this Report.

The opinions expressed in this Report are appropriate as of the Effective Date. The Report is only appropriate for this date and may change in time in response to variations in economic, market, legal or political factors, in addition to ongoing exploration results.

2.11 **Consent**

SGS consents to this ISR being included in a prospectus issued by the Company and distributed, in full, in the form and context in which it is provided.

SGS provides this consent on the understanding that the Technical Assessment expressed in the individual sections of this ISR will be considered with, and not independently of, the information set out in full in this report.

2.12 Units and Abbreviations

All units of measurement used in this technical report are in metric. All currency is in US dollars (US$), unless otherwise noted.

Table 2-2 List of Abbreviations

t or Tonnes	Metric tonnes	Sc	Scandium
Mt	Million tonnes	Y	Yttrium
tpd	Tonnes per day	La	Lanthanum
kg	Kilograms	Ce	Cerium
g	Grams	Pr	Praseodymium
NSR	Net Smelter Return	Nd	Neodymium
M	Metres	Pm	Promethium
cm	centimetre	Sm	Samarium
mm	millimetre	Eu	Europium
Km	Kilometre	Gd	Gadolinium
L	Litre	Tb	Terbium
mL	Millilitre	Dy	Dysprosium
ppm	Parts per million	Ho	Holmium
N	North	Er	Erbium
E	East	Tm	Thallium
S	South	Yb	Ytterbium
W	West	Lu	Lutetium
ha	Hectare	Hf	Hafnium
m^3	Cubic metres	Ta	Tantalum
CAD$	Canadian Dollars	W	Tungsten
MGA	Malagasy Ariary	Th	Thorium
QA	Quality Analysis	U	Uranium
QC	Quality Control	Zr	Zirconium
%	Percent	Nb	Niobium
°	Degrees	Ti	Titanium
°C	Degrees Celsius	In	Indium
CoG	Cut-Off Grade	Be	Beryllium
TREE	Trace Rare Earth Elements	TREO	Trace Rare Earth Oxides
HREO	Heavy Rare Earth Oxides	CREO	Critical Rare Earth Oxides
LREE	Light Rare Earth Elements	HREE	Heavy Rare Earth Element

3 HARENA RESOURCES PTY LTD

Harena Resources is an Australia-based, exploration and development company, focused on the development of critical rare earth element mineral projects. The Company was originally incorporated as Harena Resources Pty Ltd in Australia on 21 April 2022 under the Australian Corporations Act (2001). At the time of writing, the Company's key asset is a 75% ownership of the development-stage Ionic Clay rare earth element ("REE") Ampasindava deposit located in Antsiranana Province, Madagascar.

The Company is headquartered in Perth, Australia, and the shares of the Company are currently privately held. The Company is subject to an offer from London listed exploration Company Citius Resources PLC (LSE:CRES) with its primary rare earth element mineral asset in Madagascar considered to be prospective for future development.

3.1 Business Strategy

After being acquired by CREC, the Company intends to increase shareholder value by utilizing the proceeds generated from a fundraising to continue with the development of the Ionic Clay REE Ampasindava deposit in Madagascar. Initial development activities will consist of the commencement of an accelerated Feasibility Study ("FS"), a new Environmental and Social Impact Assessment ("ESIA") Plan, validation of metallurgical test work with new samples, and the engineering design and planning for a Demonstration Plant.

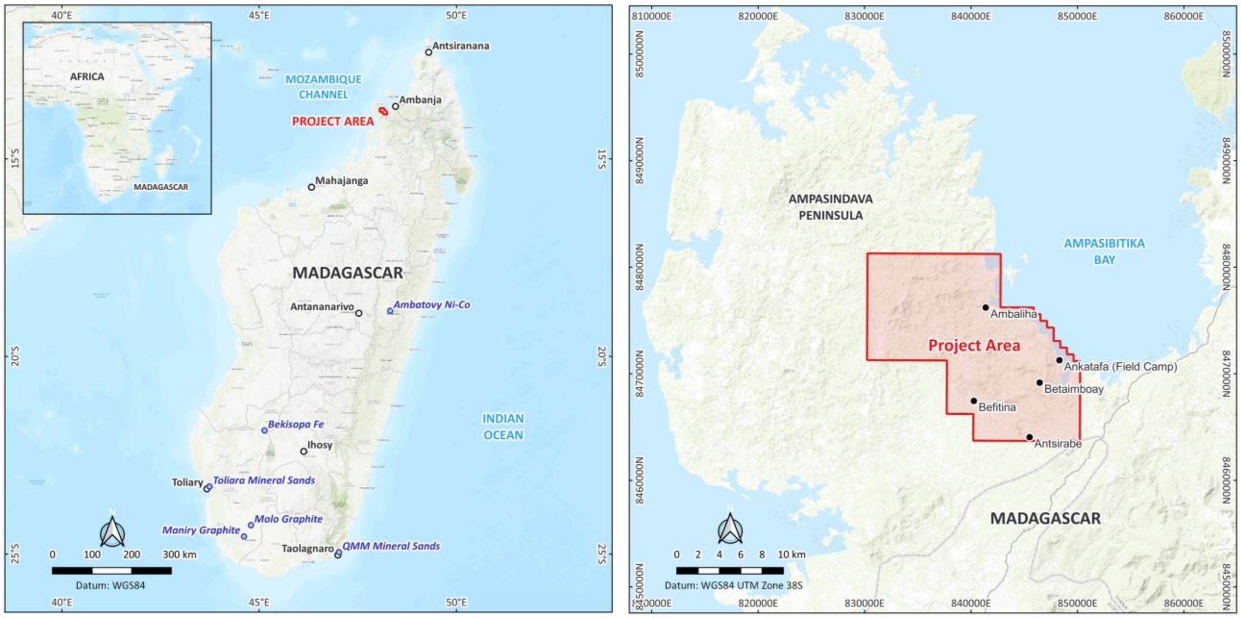
4 PROJECT SETTING

4.1 Location and Access

The Ampasindava project is located in the eastern part of the Ampasindava Peninsula, Antsiranana Province on the northwest coast of Madagascar, approximately 500 km north of Madagascar's capital city Antananarivo (Figure 4-1). The nearest major town and administrative centre of the region is called Ambanja and is located some 40 km to the northeast of the project area.

The Property is centered at approximately 13.8421°S latitude, 48.1459°E. Alternatively the coordinates in WGS84 UTM Zone 38S are 191,500 m E; 8,467,900 m N, and coordinates in the local Laborde coordinate system are 584,898 (X), 1,358,752 (Y).

Figure 4-1 Project Location Map

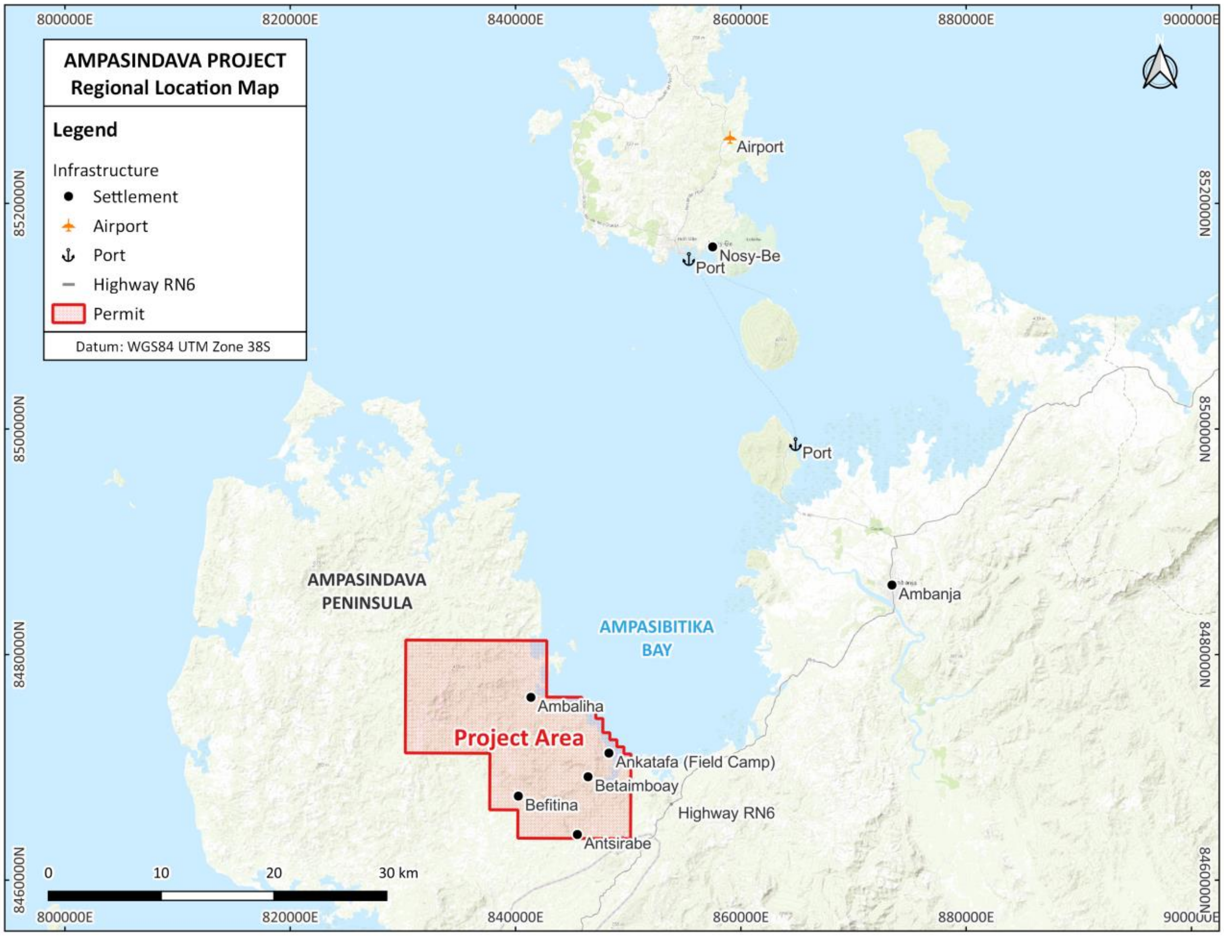


The nearest international airport to the project area is Fascene, located on the island of Nosy Be (Figure 4-2). Airlines that currently operate include Air Madagascar, Air Austral and Air Italy with destinations including Antananarivo, La Reunion, Mauritius, Johannesburg, Milan and Rome.

Access from Nosy Be to the project area is by boat and Harena Resources has its own craft for this purpose. The travel time from the port of Madirokely in the southwest of Nosy Be to the project area is approximately 50 minutes, corresponding to a distance of approximately 40 kilometres.

Road access to the project area requires the use of a 4×4 vehicle along a purpose-built track that connects to the main Route Nationale 6 (N6) highway approximately 30 km southwest of Ambanja. The main highway passes within 2 km of the southeast corner of the project area (Figure 4-2). Vehicular access around the project area is limited to a few dirt tracks. These are passable using 4×4 vehicles only and restricted to dry conditions. Most access around the project area is on foot.

Figure 4-2 Regional Project Location Map



4.2 Climate and Physiography

The climate in Madagascar can be broadly divided into two distinct seasons: a dry season and a wet season. The dry season typically occurs between April and October and the wet season from November to March. The rainy season is generally very wet and accompanied by high temperatures on account of the eastern trade winds and cyclonic influence. Typically, the weather is warm all year round due to the country’s position astride the Tropic of Capricorn. However western coastal areas can become very hot during the summer dry season. Annual rainfalls are more than 3000 mm in the eastern coastal plains, around 1500 mm in the central plateau and less than 500 mm in the western coastal plains.

The Ampasindava project area is associated with an average annual temperature of greater than 25°C and rainfall exceeding 2000 mm per year, conditions that are conducive to the weathering of the bedrock and the formation of regolith.

The majority of the project area is relatively rugged with elevations ranging from sea-level to 713 m with the highest elevations found in the northwest of the project area. The rugged terrain can make access to certain parts of the project area problematic, particularly in the rainy season. The most characteristic physiographical feature in the project area is a 6 km wide, circular caldera which corresponds to the southeast part of the Ambohimirahavavy igneous complex (Figure 4-3) (Gilbertson 2013).

Figure 4-3 Photograph of the Ambohimirahavavy Igneous Complex Caldera (Gilbertson, 2013)

The majority of the project area is covered by secondary vegetation including bamboo, traveller's palms and other species. The original primary forest is restricted to a few mountain tops and a small area in the extreme northwest. Original primary forest covers less than 20 km^2 of the 237.5 km^2 project area. Please note that no sampling was done in the primary forest areas and therefore this area is not included in the mineral resource statement. Malagasy environmental legislation protecting these areas does not restrict exploration activities in the vast majority of the project area, but it was decided by previous operators to leave them undisturbed. Shallow tidal areas in bays in the coastal areas are covered by mangroves.

Slash and burn agriculture is very common throughout much of the permit area, increasingly evident as areas of barren ground (Figure 4-4).

Figure 4-4 Photographs of a General View of the Eastern Part of the Ampasindava Project Taken in December 2013.

Note: Top image shows a sector burned to prepare for planting rice with a bamboo stand in the bottom of the valley. Bottom image is taken from the eastern limit of the caldera showing a patchwork of vegetation from the rotating agriculture usage.

4.3 Local Resources and Infrastructure

Ambanja represents the logistical centre of the region with infrastructure that includes a hospital, banks, restaurants, hotels and courier services, and so forth. There is only very limited infrastructure within the project area. Four semi-permanent field camps, including the Ankatafa Camp (Figure 4-2), were previously constructed in strategic locations to support exploration activities.

Mobile telephone networks are available in parts of the project area and at the Ankatafa camp, but signal reception is sporadic and weak. Internet access is only possible through the mobile network.

4.4 Mining Industry in Madagascar

Madagascar has good potential to discover and develop new deposits for a range of commodities, although its mining industry remains relatively underdeveloped. The country currently has mines producing nickel, chromium, cobalt and ilmenite ores and has other large deposits containing gold, nickel-cobalt, heavy mineral sands (titanium), bauxite, copper, lead, manganese, platinum, zinc, zirconium, coal and petroleum products. It is noted for its production of good quality chemical and metallurgical grade chromite, high-grade crystalline flake graphite and mica, and has an abundance of semiprecious stone deposits.

In 2016, Madagascar played a significant role in the world's production of cobalt, ilmenite, mica, and nickel. The country's share of world production of mica amounted to about 6%; mined and refined cobalt; 3% each; and ilmenite and mined nickel, 2% each (USGS, 2021). Production from the Ambatovy nickel-cobalt project, run by the mining company Ambatovy (Sumitomo Corporation) ranks among the largest in the world (EITI, 2023). Other notable projects include the large ilmenite and mineral sands mine run by QIT Madagascar Minerals (Rio Tinto) at Fort Dauphin, and the Molo Graphite mine in the province of Toliara run by NextSource Materials.

In 2018, Madagascar's mining sector accounted for 4.6% of GDP, 4.4% of total government revenues, and 28% of total exports. The country is also considered to be a new frontier for oil and gas prospecting, but oil exploration remains limited (EITI, 2023).

In line with its overall policy defined in 1998 in the Document Cadre de Politique Economique (DCPE), the five-year Mining Sector Reform Project (MSRP), led in part by the World Bank Group, assisted the Government in setting up a legal and regulatory framework conducive to private investment in the area of mineral resources with the aim of attracting large-scale mining projects. Another key objective was to shift the role of the State from operator to regulator and promoter of sustainable minerals development. Many of the World's economic development agencies such as USAID, International Monetary Fund and World Bank Group committed significant investments and resources to improve the sector.

Reforms, supported by the MSRP, include:

1. a new Mining Code and its regulations, which have established an adequate legal and regulatory framework to attract private investment into mining, including environmental regulations for mining, published jointly by the Ministry of Environment and the Ministry of Energy and Mines;
2. a special law for large-scale mining investments, defining an attractive special investment regime for mining in Madagascar, and providing for a fair share of revenues between the central and provincial Governments and the private sector; and
3. improved governance through the establishment of the Mining Cadastre, a non-discretionary and transparent system to grant, manage and cancel mining permits.

The most recent reforms to the Madagascar Mining Code were ratified in 2005. Subsequent reform of this legislation began in 2019 and at time if writing in 2023 the revised draft Ming Code legislation had been signed by Madagascar's President Andry Rajoelina and awaits ratification by the Senate. This Mining Code update has resulted from the Malagasy Council of Ministers approving the Ministry of Mines and Strategic Resources to resume processing applications for all mining permits on March 30, 2023. Key changes to the Mining Code include the Mining Royalty increased from 2% to 5% and the introduction of a 3% Social and Community Mining Investment fund, based on each project's capital cost.

Recent Validation by the global Extractive Industries Transparency Initiative (EITI) of Madagascar's extractive sector relative to the 2016 EITI Standard, a global standard for the good governance of oil, gas and mineral resources, found the country to have made 'Meaningful Progress' in June 2020. The country's next Validation, relation to the EITI Standard, was expected to commence in April 2023 (EITI, 2023).

In summary there are some significant projects at advanced or development stages in Madagascar and there appears to be willingness from foreign companies to invest in large projects. This has no doubt been helped by the recent reforms of the Malagasy Mining Code.

However, the main factors contributing to the underdevelopment in the mining sector include the need for major infrastructure upgrades, its poor electrical power distribution systems, under-funded health and education facilities, difficulties in reforming the economy and dealing with chronic malnutrition, deforestation, land erosion and population growth.

4.5 Ownership, Land Access, and Tenure

The Cadastral Mining Office of Madagascar, BCMM, Ministry of Mines and Strategic Resources, grants subsoil use rights and manages mining licenses. New licenses are normally issued on a first come first served basis. However, the issuance of new licenses in the mining sector has been suspended by the government for several years (EITI, 2023).

The Ampasindava Project comprises one exploration licence (permit PR 6698) made up of 608 contiguous 625 m by 625 m unit blocks that encompass a total area of 237.5 km^2 (Figure 4-5). The permit is currently granted as a "Permis de Recherche" (research permit), exploration licence, or PR, which grants the exclusive right for prospecting and research.

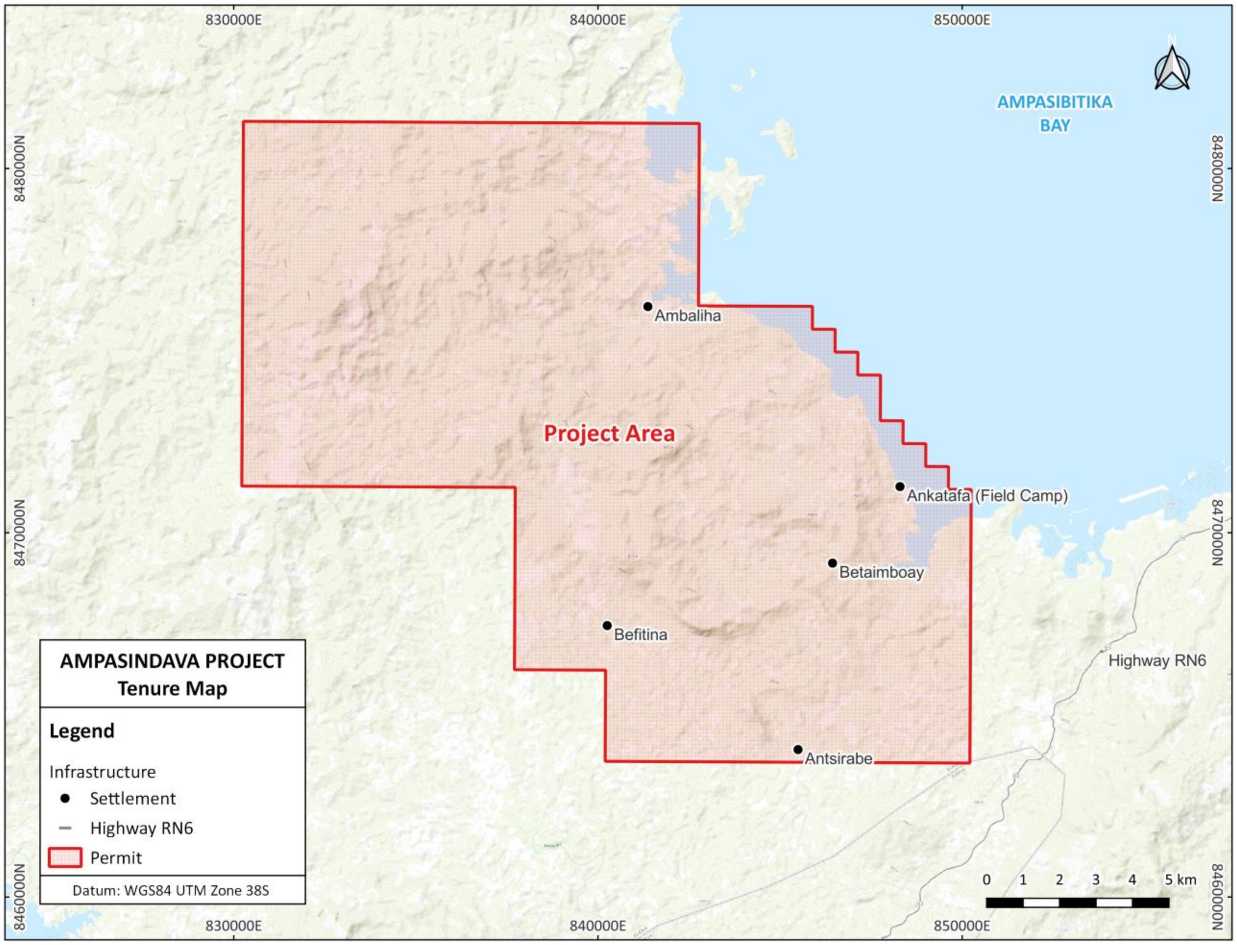
The permit was originally granted in 2003, under the regime of the 1999 Mining Code for a period of 10 years with the provision for two additional 5-year term extensions. An amended Mining Code came into force in 2005 with the resultant change being that term renewals of PR 6698 would be for 3-year terms, with renewal terms initiating upon the mining registrar signing date. The first permit renewal application was in 2012 (signed in 2014) and the second in 2016 (signed in 2018) (MCI, 2023).

In keeping with article 33 of the 2005 Mining Code, the exploration licence (PR 6698) cannot be renewed for a third time; however, it can be converted to an exploitation licence (PE). Exploitation licences provide the tenure holder with the right to mineral extraction and concomitant exploration for an initial term of 40 years, with provisions for term renewal. A valid application for a permit conversion of PR 6698 to an exploitation license was filed on September 18, 2020 (MCI, 2023). At the Effective Date of this Report the permit conversion procedure remains in process.

The permit was originally held by Calibra Resources and Engineers Madagascar SARL and was subsequently acquired by Zebu Metals Limited in January 2008. Tantalus Rare Earth AG acquired Zebu Metals and the Project in October 2009. The Project was previously known as the Tantalus Rare Earth Ionic Clay project.

The current 100% beneficial owner of exploration licence PR 6698, currently under conversion to an exploitation licence, is Reenova Rare Earth (Malagasy) Sarl.u ("RREM"), a Malagasy company. Harena Resources Pty Ltd have acquired the parent company of RREM to hold a 75% interest in the Property.

Figure 4-5 Claim Map for the Exploration Licence PR6698 (Ampasindava Project)



4.5.1 Other Land Uses

Harena Resources is required to continue engaging with the local communities (many small communes) and local landowners to gain access to exploration targets and for the development of project infrastructure. The Company is obligated by law to get permission from landowners prior to doing any exploration work. Previous operators have engaged in discussions with local and national governments to establish the framework for continued work on these lands and Herena Resources is advised to continue with this engagement process.

4.5.2 Royalties

There are no known royalties on the project.

4.6 Project History

An overview of the Ampasindava Project exploration history is listed in Table 4-1.

Table 4-1 Ampasindava Project Exploration History

Ampasindava Project Exploration History	
2003	Exploration licence (PR 6698) originally acquired by Calibra Resources and Engineers
2008	In January 2008 Zebu Metals acquired the Project from Calibra Resources and Engineers; Stream and beach sediment sampling (5), one trench excavated that confirmed significant REE mineralisation (up to 0.2% TREO), mini bulk samples (2) of granitic intrusive for geochemical analysis, airborne magnetic and radiometric surveys
2009	Geological interpretation of the magnetic and radiometric surveys; In October 2009 Tantalus Rare Earths AG acquired Zebu Metals and the Project
2010	Mineralogical test work focused on regolith-hosted ionic adsorption-type REE mineralisation
2010-2011	Diamond Drilling – 277 holes (NW, NTW, and BTW) completed to test for the presence of bedrock-hosted REE mineralisation
2012	Initiation of metallurgical test work regolith-hosted ionic adsorption-type REE mineralisation
2011-2013	Pitting – 4,474 manually excavated pits dug to assess regolith- hosted REE mineralisation
2013	Initial Mineral Resource Estimate completed by SRK
2013-2014	Advanced metallurgical test work to assess the amenability of the deposit to leach processing methods
2014	Updated Mineral Resource Estimate completed by SGS

4.7 Regional Geological Setting

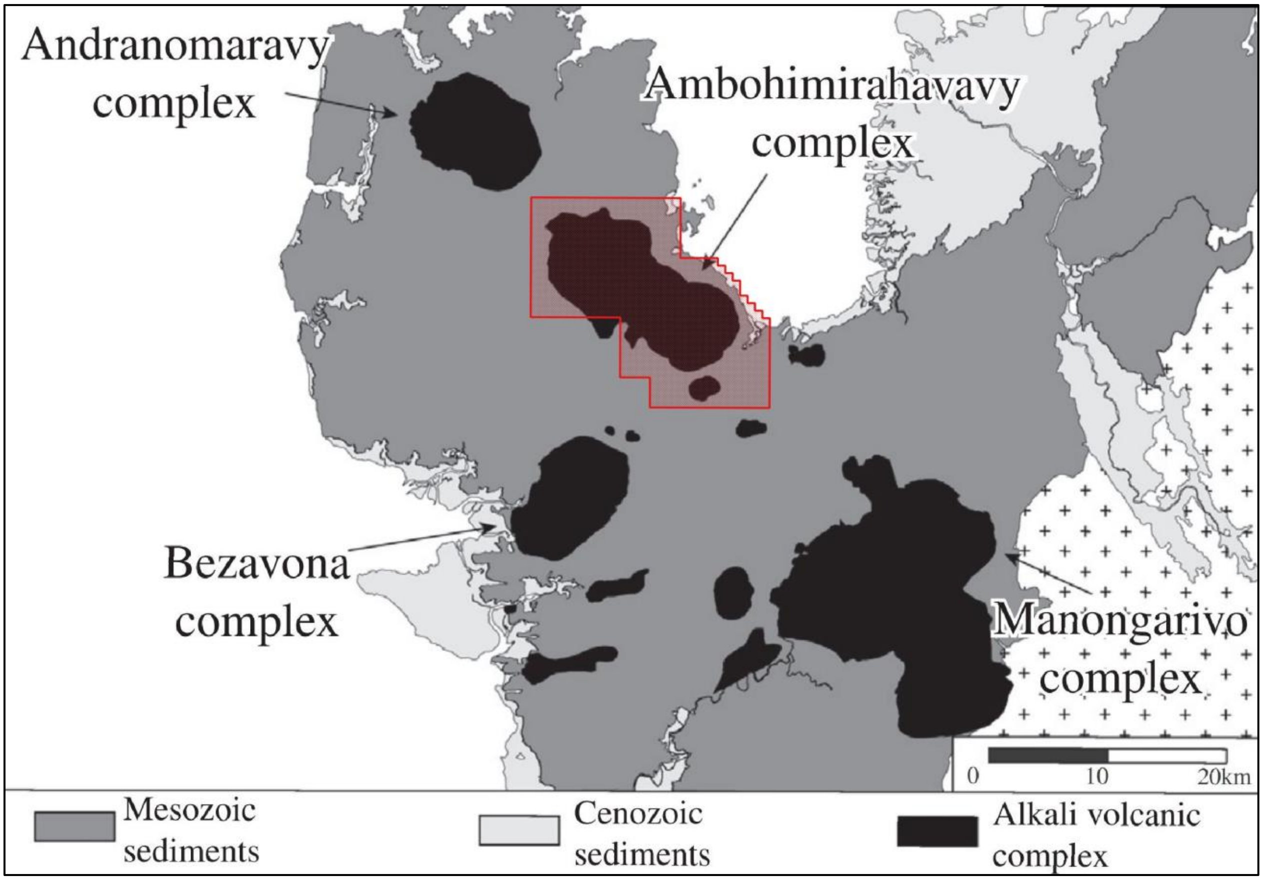
Northern Madagascar is dominated by Mesozoic sediments that were deposited in a predominantly marine environment and include mudstone, siltstone, limestone, sandstone and marl.

In the late Cenozoic, the central and northern parts of Madagascar were subject to uplift and rifting that resulted in the development of horst and graben structures. This extensional regime was also accompanied by intra-continental volcanism and the emplacement of numerous igneous complexes, including several that occurred along a roughly linear southeast-northwest trending zone between the Nosy Be archipelago and Antongil Bay. The chronology of the emplacement of the igneous complexes is poorly constrained, but thought to have occurred between the Eocene and Late Miocene (Ganzeev and Grechishchev, 2003 and Melluso, et al., 2007).

The igneous rocks are very diverse and range in composition from mafic-ultramafic (olivine melilitite, olivine nephelinite, basanite, tephrite, alkali basalt and hawaiite) to intermediate (tephritic phonolite and phonolite) to acidic (quartz trachyte and rhyolite).

In the region of interest the igneous rocks form part of what is called the Ampasindava alkali-bearing province that predominantly occupies the Ampasindava peninsula (Figure 4-6). The Ampasindava igneous rocks occur as massifs and include alkali syenite, foid syenite, alkali granite, gabbro, alkali trachyte, phololite, rhyolite and volcanic breccia. One of these massifs is called the Ambohimirahavavy igneous complex and occurs almost entirely within the Ampasindava project area.

Figure 4-6 Simplified Geological Setting of the Ampasindava Peninsula (after Estrade et al, 2013), Project area in Red.



4.7.1 Deposit Model

4.7.1.1 Deposit Summary

The Ampasindava project area is underlain by the Ambohimirahavavy igneous complex that encompasses an area of approximately 150 km^2. Significantly, it includes alkaline and peralkaline rocks that are mineralised with REE and other rare metals (including tantalum, niobium and zirconium).

The presence of a favourable sub-tropical climate has resulted in the development of widespread regolith. Based upon the available drilling and test pitting data, the thickness of the regolith averages approximately 13.5 m, but has attained thicknesses of greater than 40 m. The regolith profile is well- developed and comprises recognised subdivisions that include soil, ferruginous, mottled and pallid zones, saprolite and saprock. The distribution of REE mineralisation within the regolith profile is erratic, but generally increases with depth. Preliminary mineralogical and metallurgical testwork has confirmed the presence of REEs that are ionically-adsorbed onto clay minerals. The testwork has also proven that the REEs can be recovered using comparatively inert solutions that include sodium chloride and ammonium sulphate.

4.7.1.2 Geological Characteristics

The REE-enriched source rocks of the Ambohimirahavavy igneous complex were emplaced as a result of intra-continental extension that induced rifting and related volcanism. The main source rocks are alkaline and peralkaline granitic dykes and sills (locally and historically termed fasibitikite), but also includes the more fractionated parts of the complex. Major southeast-northwest structures appear to have imposed a fundamental control on the location of the complex.

Subordinate structures may have influenced the location of the mineralised dykes and sills and acted as preferential pathways for post-intrusive hydrothermal fluids. The mineralised source rocks were subject to intense weathering due to the sub-tropical climate (average temperatures of higher than 25°C and rainfall exceeding 2000 mm per year) that resulted in the development of widespread and typically thick regolith.

4.7.1.3 Mineralisation Characteristics

Geometrically the regolith profile in the Ampasindava project area ranges in thickness from 0 m to more than 40 m. Based upon the available drilling and test pitting data, the average thickness is approximately 13.5 m. The profile is well-developed and includes recognised subdivisions. The two primary subdivisions are the pedolith and the saprolite. Secondary subdivisions of the pedolith (from the surface downwards) include soil, a ferruginous zone, a mottled zone and a pallid zone. Secondary subdivisions of the saprolite include saprolite and saprock.

The entire regolith profile contains REE mineralisation, but its distribution is typically quite erratic. Despite this, general trends are present with REE content typically increasing with depth and then decreases approaching the un-weathered bedrock. This trend also corresponds to the enrichment of HREEs relative to LREEs with depth. The two fundamental controls on the formation of ion adsorption REE mineralisation are the availability of an REE-enriched source rock and in-situ sub-tropical weathering conditions that enable the liberation and mobilisation of the REEs and their preferential adsorption onto the surfaces of clay minerals. Both of these essential components are present in the Ampasindava project area.

4.8 Commodity Background and Markets

4.8.1 Critical Rare Earth Elements

The rare earth elements (REE), sometimes referred to as the rare earth metals, are a family of 17 chemically similar metallic elements comprising 15 elements in the lanthanides group, plus scandium and yttrium. The lanthanides are elements with atomic numbers 57 to 71 and comprise lanthanum (La), cerium (Ce), praseodymium (Pr), neodymium (Nd), promethium (Pm), samarium (Sm), europium (Eu), gadolinium (Gd), terbium (Tb), dysprosium (Dy), holmium (Ho), erbium (Er), thulium (Tm), ytterbium (Yb) and lutetium (Lu). Scandium (Sc) and yttrium (Y) are considered REE as they have similar chemical properties to the lanthanides.

The rare earth elements are commonly divided into light rare earth elements (LREE) and heavy rare earth elements (HREE) on the basis in their atomic numbers. However, the formal definition of what constitutes LREE or HREE is not consistent. In the context of this report, and consistent with the majority of published definitions, LREE include rare earth elements with atomic numbers between 57 and 62 (i.e. La, Ce, Pr, Nd, Pm and Sm). HREE include rare earth elements with an atomic number of 63 or greater (i.e. Eu, Gd, Tb, Dy, Ho, Er, Th, Yb and Lu) plus Y.

Rare earth elements do not occur naturally as metallic elements, they occur in a range of minerals that include carbonates, halides, oxides and phosphates. A total of approximately 200 REE minerals have been identified.

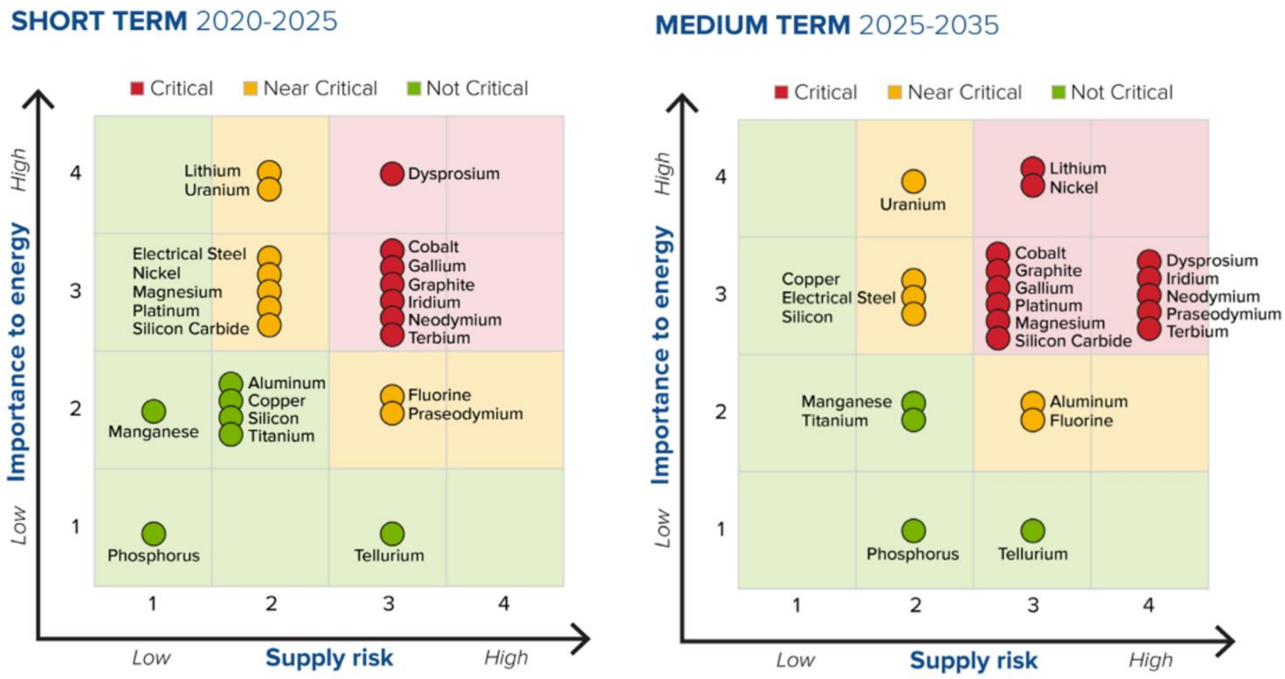
While each REE is used in different applications, four elements – neodymium, dysprosium, praseodymium, and terbium – are of particular importance to the clean energy sector (EIA, 2021). Since the mid-1990s, China had emerged as a major producer. Its share of global production rose to over 95% in 2010, since then its share has fallen to just over 60% in 2019, as the United States, Myanmar and Australia started to boost production (USGS, 2021). However, separation and refining operations are still heavily concentrated in China, with almost 90% market share in 2019. There are currently four plants operating outside China.

These plants, however, process only light REEs and the processing of heavy REEs is entirely dominated by China (EIA, 2021).

China's attempt to limit REE exports in 2010 triggered many countries to consider options to reduce material intensity, find substitutes and diversify sources of production. Some 20 projects are under development in Australia, Canada and the United States, of which 5 projects plan to start operations in the early 2020s. Several processing plants are also under development, most notably in the United States (EIA, 2021).

In the 2023 Critical Materials Report from the U.S. Department of Energy, it was reported that three of the 17 rare earth elements were found to be critical in terms of supply risk in the short term (2020 to 2025), with four rare earth elements identified as critical into the medium term (2025 – 2035) (Figure 4-7). These elements are termed "critical" because of their scarcity. This, combined with anticipated growth in demand, makes these higher valued elements the ones expected to experience the best price performance over the next decade. As shown in the criticality matrix below, those elements deemed critical over the next decade are: dysprosium, neodymium, praseodymium, and terbium. These four rare earth elements are used in magnets for wind turbines and electric vehicles or phosphors in energy-efficient lighting.

Figure 4-7 2023 Short- and Medium-Term Mineral Criticality Matrices (US Department of Energy, 2023)



The Ampasindava Ionic clay project displays a pervasive and well-balanced rare earth distribution that is prevalent throughout the deposit. The deposit includes appreciable amounts of the critical rare earths dysprosium, neodymium, praseodymium, and terbium as defined by the U.S. Department of Energy.

The Ampasindava Project is one of relatively few projects of ionic clay hosted REE deposits outside of China. This sets it apart from most of the developers vying to be a stable source of these metals (particularly the HREE) for the western world.

5 AMPASINDAVA PROJECT

5.1 Overview

The Ampasindava Project was first recognised for hosting mineralised dykes hosting high-grade Rare Earth Elements (REE) in fresh rock. REE hosted within the regolith (or soil profile) was recognised in 2012; and it has become clear that this “ionic clay” style of mineralization has the most significant economic potential. The ionic clays are most important mineralization type for the Project due to the relatively simple process required to put the REE into solution and the fact that neither thorium nor uranium is concentrated through this process; this is in stark contrast to most REE deposits being developed or promoted in the western world. Additionally, the geological setting of the Ampasindava Project is analogous to the source of most of the Heavy REE (HREE) currently produced in China.

5.2 Project Geological Setting

5.2.1 Lithology

The Project area is underlain by Jurassic sediments into which the Ambohimirahavavy igneous complex has intruded. The Jurassic Isalo Group sediments are dominated by mudstones and siltstones that are interbedded with sandstones, marls and minor limestone. They comprise an estimated thickness of approximately 2500 m and dip westwards between 5° and 30° (Ganzeev and Grechishchev, 2003). Aside from localised skarn development adjacent to some of the intrusive rocks, the sediments are un-metamorphosed.

The crudely oval Tertiary Ambohimirahavavy igneous complex is approximately 20 km in length, up to 8 km in width, elongated in a southeast-northwest orientation and encompasses an area of approximately 150 km^2. The complex consists of two arcuate intrusions comprising predominantly syenites known as the Ampasibitika intrusion in the southeast and the Tsarabariabe intrusion in the northwest. These intrusions are characterised by central depressions that are interpreted to be calderas and include volcanic rocks of predominantly trachyte composition. Several smaller intrusions (several hundreds of metres across) of alkali granite and alkali quartz syenite occur within the complex.

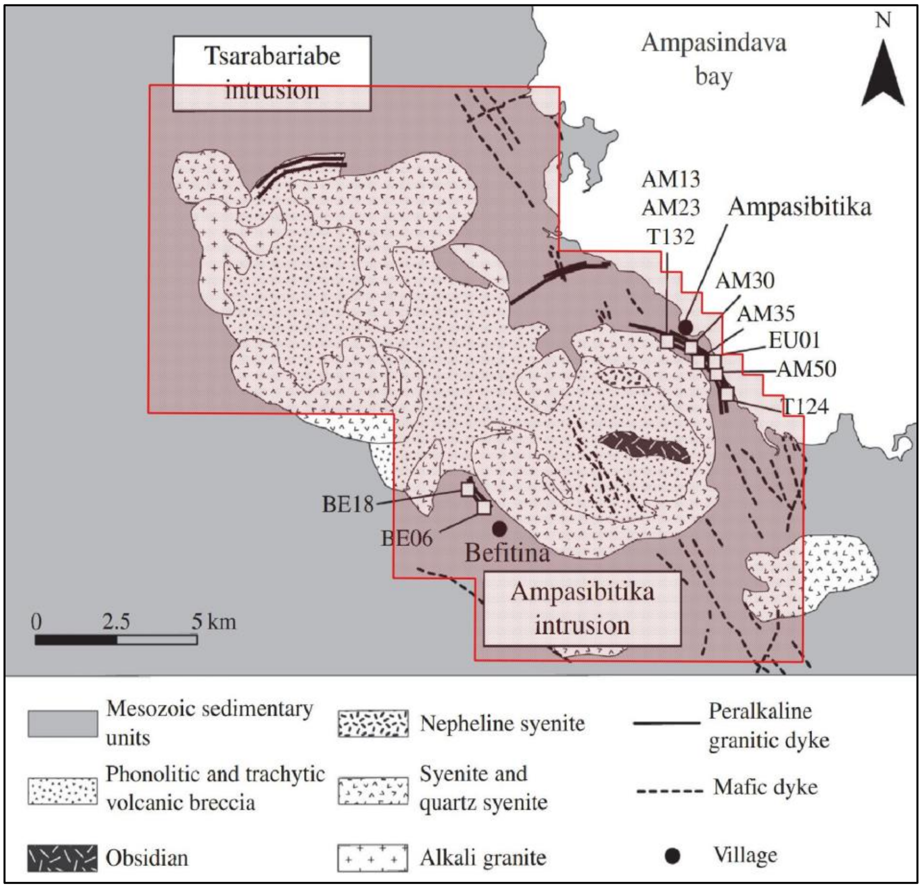
Based upon an interpretation of available geophysical data (airborne magnetic survey and radiometric survey), the Ampasibitika intrusion is near-circular with a diameter of 7.2 km (Earthmaps Consulting, 2009). Magnetic data shows that it has a well-defined outer rim of magnetic syenite and an inner, almost circular, core of non-magnetic granite/rhyolite and syenite. The magnetic syenite is more resistant to weathering and forms high terrain, while the non-magnetic granite and syenite are more susceptible to weathering and form low terrain in the centre of the intrusion. The exception is a small central rhyolite pipe which forms a cone of high terrain in the centre of the intrusion. However the rhyolite cannot be distinguished from the non-magnetic granite and syenite in the magnetic or radiometric survey data.

The Tsarabariabe intrusion is much larger and more complex than the Ampasibitika intrusion. It measures approximately 8 km by 12 km and consists of several different intrusions of which at least four can be discerned in the geophysical data. Within the Tsarabariabe intrusion the correlation of magnetic syenite and higher topographic terrain still broadly hold. However it is less consistent than in the Ampasibitika intrusion. The intrusive centre mapped as strongly magnetic syenite occupies low topographic terrain, as do the much smaller, strongly magnetic syenites northeast of the igneous complex.

Together with the Ampasibitika intrusion a total of seven distinct intrusives have been interpreted from the geophysical data (Figure 5-2). The published 1:100,000 scale geological map for the Project area is provided in Figure 5-1.

Associated with and often bounding the igneous complex are a variety of dykes and sills that have also intruded the Isalo sediments. These have compositions that include trachyte, microsyenite, trachyphonolite and granite. The peralkaline granitic varieties, locally termed fasibitikite (Lacroix, 1922), are particularly significant from an economic perspective as they contain rare earth elements and other “rare metal“ mineralisation, including tantalum (Ta), niobium (Nb), zirconium (Zr), hafnium (Hf), gallium (Ga), uranium (U), thorium (Th) and tin (Sn).

Figure 5-1 Map Showing the Geological Setting for the Ampasindava Project Area (after Estrade et al, 2013), Project area in Red.



The peralkaline granitic dykes and sills are best exposed between the Joja and Ankobabe Rivers in the vicinity of Ampasibitika village where they have been delineated over an area approximately 300 m wide and intermittently up to 8 km along strike. The southeast-northwest strike of the dykes and sills corresponds to the contact of a large semi-circular alkali-syenite intrusion (the Ampasibitika intrusion). This contact dips westward approximately 40° and obliquely cuts the adjacent sedimentary rocks (Figure 5-3). All of the Ambohimirahavavy igneous complex syenites are coarse-grained to pegmatitic and composed of idiomorphic microperthite K-feldspar and strongly xenomorphic subalkalic amphibole.

The mineralised peralkaline granitic intrusives generally dip between 15° and 55° towards the igneous complex and their thicknesses range from a few millimetres to over 15 m, although are more typically between 0.1 m and 2.5 m thick. The dykes and sills often have quite complex morphologies with pinches, swells and branches and have zonal internal structures. They can occur as a series of stacked intrusives but in places they are observed to be anastomosing and with very erratic orientations, having followed pre-existing discontinuities in the country rock. Where the intrusives have intruded calcareous country rocks there is localised skarn development, but where they have intruded other types of sedimentary rock no alteration is evident. Intrusion into larger trachyte bodies has resulted in localised and weak fenitisation.

Studies completed by the University of Toulouse have confirmed that the primary magmatic assemblage within the peralkaline granitic dykes and sills includes alkali feldspar, arfvedsonite (a variety of sodium amphibole), aegirine (a variety of clinopyroxene) and quartz (Estrade, 2011a). Identified accessory minerals include chevkinite, eudialyte, monazite, pyrochlore and zircon. Field studies identified three textural varieties of peralkaline granite: fine-grained, banded and pegmatitic, with the latter including large arfvedsonite crystals up to 20 cm in length.

Figure 5-2 Map Showing the Geological Interpretation for the Ampasindava Project Area (after Earthmaps Consulting, 2003)

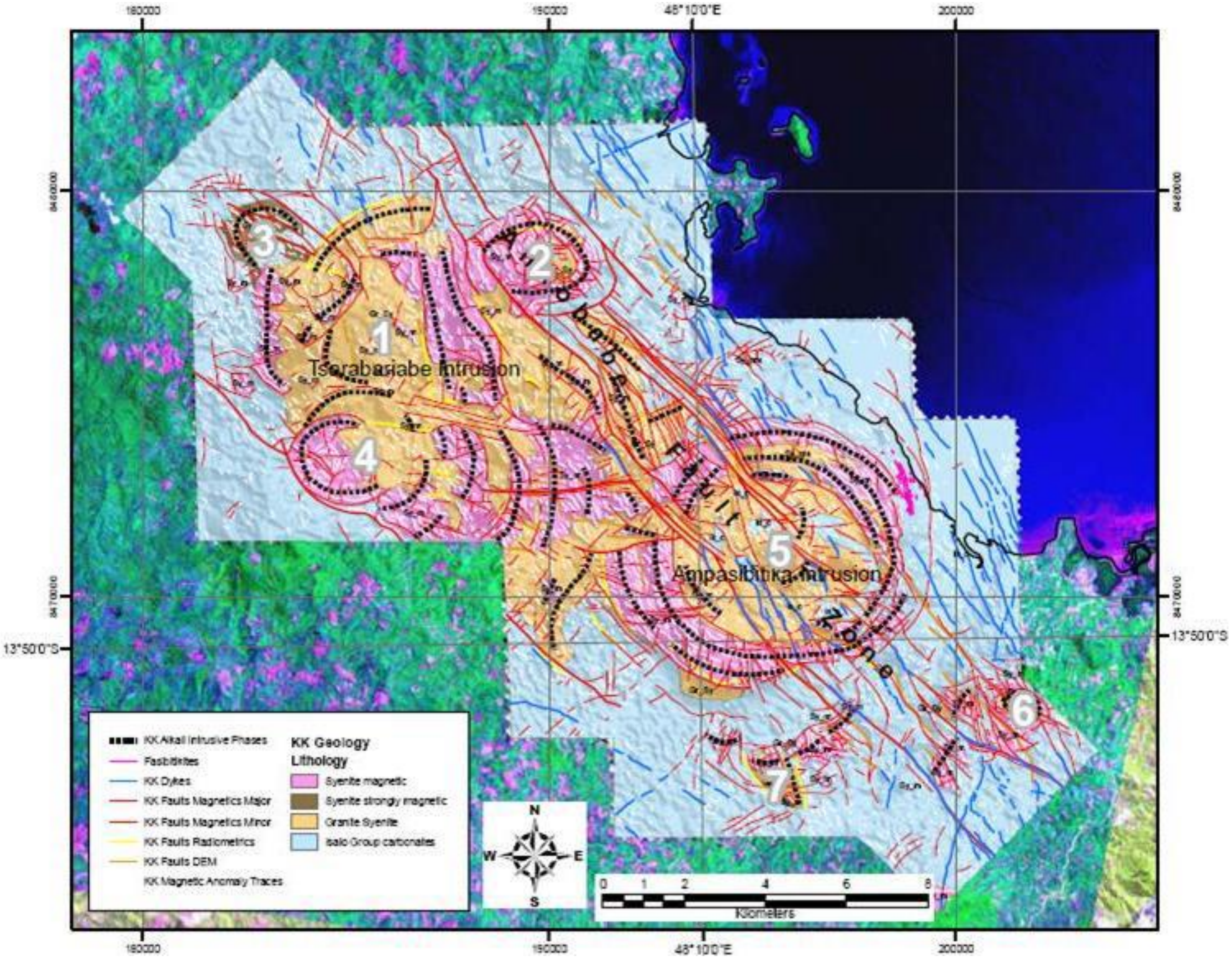
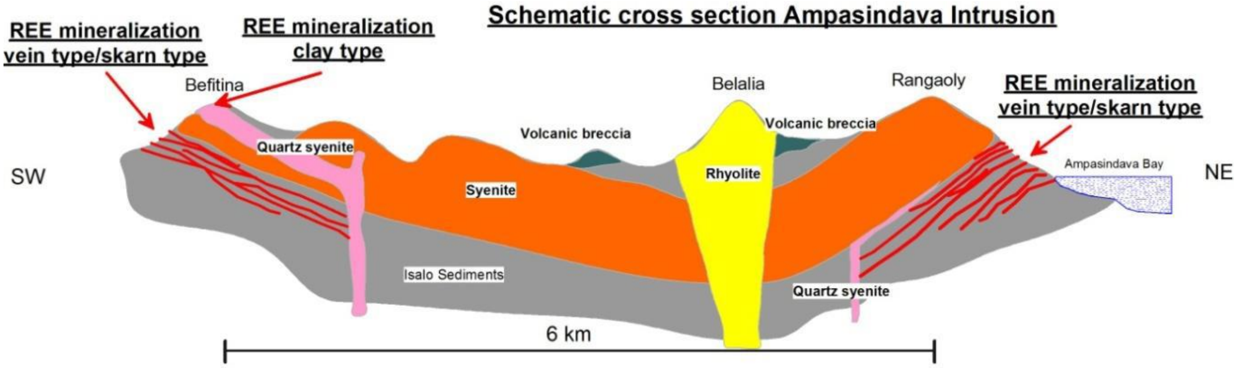


Figure 5-3 Schematic Cross-section of the Ambohimirahavavy Igneous Complex (modified from OMNIS-SM, 1992) (Gilbertson, 2013)



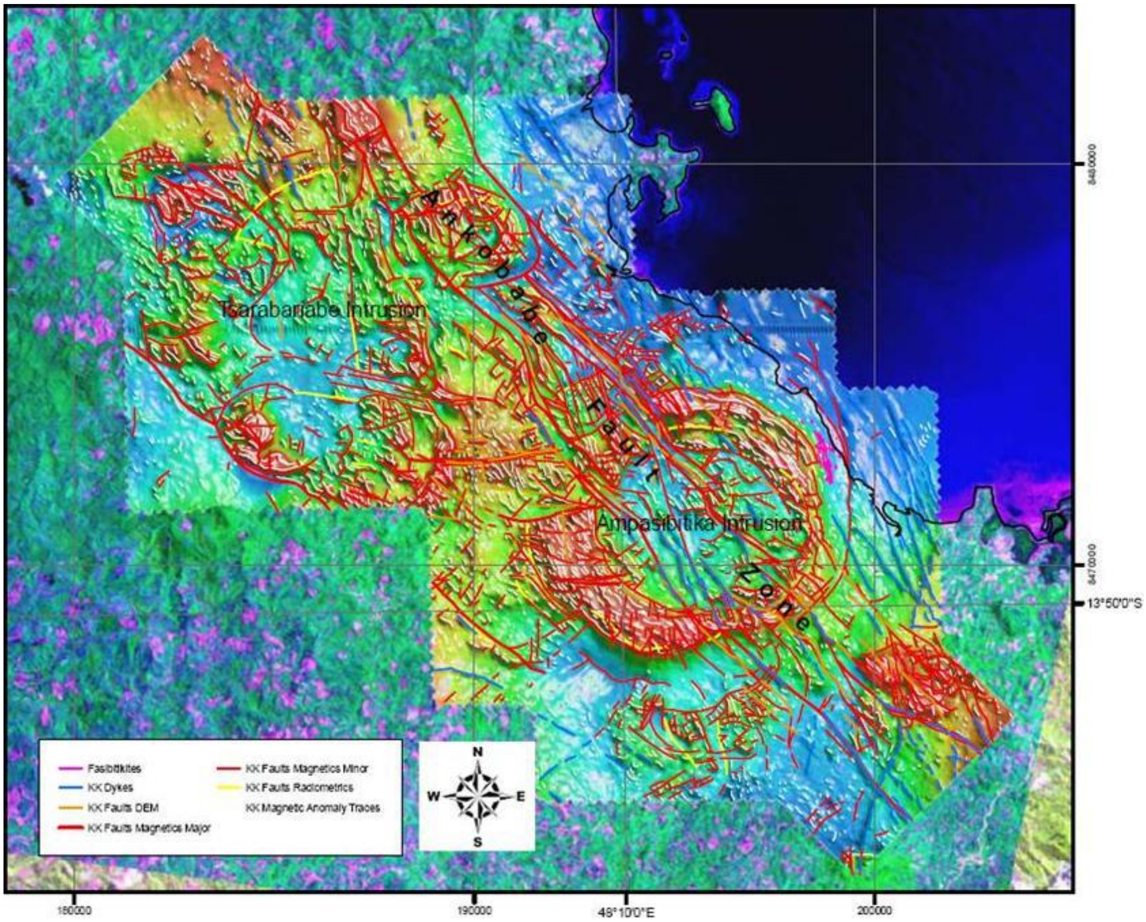
5.2.2 Structure

The most comprehensive account of the structural setting of the project area is provided by the interpretation of the available geophysical survey data (Earthmaps Consulting, 2009).

The Ambohimirahavavy igneous complex is affected by numerous structures (Figure 5-2, Figure 5-4). Both arcuate and concentric structures are evident and help to define the individual circular to sub- circular Ampasibitika and Tsarabariabe intrusive centres.

The most conspicuous structural feature is a 1.5 km to 2.5 km wide southeast-northwest trending fault zone which cuts across the north-eastern margin of the Tsarabariabe intrusion and through the Ampasibitika intrusion, hereafter called the Ankobabe Fault Zone (named after a nearby village). This fault zone is characterised by numerous sub-parallel major and minor faults clearly evident from the disruption and termination of magnetic and radiometric anomalies. This is most notable in the north- western rim of the Ampasibitika intrusion where the characteristic circular magnetic anomaly is in places completely obliterated. This may be due to alteration associated with the fault zone, or due to the intrusion of non-magnetic granites and syenites along the fault zone. Notably, the Ankobabe Fault Zone is also evident in topographical data where drainages have preferentially eroded southeast- northwest oriented incisions.

Figure 5-4 Map Showing Magnetic Imagery and Interpreted Structural Setting for the Ampasindava Project Area (after Earthmaps Consulting, 2009)



The geophysical survey data interpretation indicates that the Ankobabe Fault Zone is probably an old and deep-seated structure that pre-dates the intrusion of the Ambohimirahavavy igneous complex. It is therefore likely that it dictated the position of the igneous complex and may have been active during and possibly after emplacement. Several significant fault zones splay off the Ankobabe Fault Zone in an east-west orientation with similar disruption to the magnetic outer rim of the Ampasibitika intrusion.

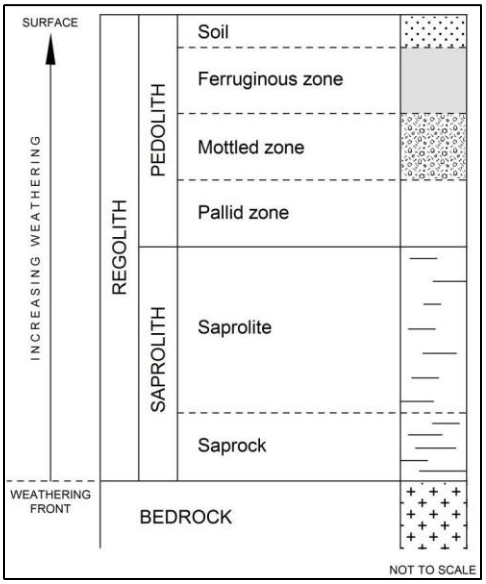
Importantly, the magnetic survey data also suggest the existence of ring faults along the edge of the Ampasibitika intrusion particularly to the north, east, south and southwest. It is these structures that may host or influence the location of the mineralised peralkaline granitic dykes and sills (Figure 5-4).

5.2.3 Regolith

Within the Project area, bedrock is largely obscured by regolith. Regolith is the term used to describe the weathered material that occurs above un-weathered bedrock and its formation is due to many different factors including climate, bedrock composition and structure, the rate of weathering, the rate of erosion, tectonic history and anthropogenic activity. Climate arguably represents the most important factor with regolith formation augmented by the presence of elevated temperature and rainfall. In northern Madagascar the climatic conditions are particularly conducive to the formation of regolith with average temperatures greater than 25°C and rainfall exceeding 2000 mm per year.

If conditions are favourable, the regolith can develop into a generally predictable profile that includes several distinct subdivisions, each with its own physical and chemical characteristics. Because of the favourable conditions, the Project area includes a well-developed regolith profile that includes the majority of recognised subdivisions (Figure 5-5).

Figure 5-5 Schematic Cross-section of the Ampasindava Project Regolith Profile (Gilbertson, 2013)



The two primary subdivisions are the pedolith and the saprolite. The pedolith can include both residual in-situ weathering products, in which all traces of the original bedrock textures and fabrics have been destroyed, and transported material such as alluvium, colluvium and aeolian deposits. Secondary subdivisions of the pedolith, from the surface downwards, include soil, a ferruginous zone, a mottled zone and a pallid zone.

Soil is difficult to define because it is used in so many different contexts by many different sectors, including agriculturalists, engineers and soil scientists. In the Project area soil is characterised by a generally thin layer, less than 0.5 m thick, of typically brown, unconsolidated, soft, homogenous material that often contains organic matter (leaves, plant roots, etc.). The most practicable way of visually differentiating soil from the underlying ferruginous zone is by colour, because it contains comparatively less iron, and by the presence of organic matter.

The ferruginous zone is characterised by the accumulation of iron and aluminium oxides, the former of which is responsible for its typical red colour. The ferruginous zone can have varying consistency ranging from being unconsolidated through to lithified, with the latter often referred to as ferricrete or duricrust. In the Project area the ferruginous zone typically comprises variable thicknesses of orange-red, soft to hard, homogenous, iron-oxide rich material. Lithified ferricrete or duricrust is characteristically absent in the Project area and this is attributed to the persistence of the rainfall and the lack of dehydration of the upper sections of the regolith profile.

The mottled zone is texturally characterised by the localised concentration of iron oxides as spots, blotches and streaks, commonly broadly rounded in outline but with diffuse boundaries. The intensity of mottling tends to decrease with depth and represents the transition with the underlying pallid zone. It is generally accepted that the mottled zone forms by weathering at or about a fluctuating water table. In the Project area the mottled zone varies in thickness and where present is usually orange- red in colour and easily distinguished by its textural heterogeneity (mottling).

The pallid zone is also often referred to as the plasmic or arenose zone. The term pallid has been adopted in the context of the Project because it more explicitly describes this part of the profile (“lacking colour”) and eliminates the genetic implications of the other two terms (the term plasmic is often used to describe this section of the profile above quartz-poor rocks, and arenose is used to describe it above quartz-rich rocks).

The pallid zone represents the transition between the mottled zone and saprolite and as its name suggests it is typically pale in colour due to a low iron-oxide and higher clay content. In the Project area, the pallid zone varies in thickness and is characterised by light, buff-coloured, firm to hard, homogenous clay-dominant material. The absence of mottling and a primary fabric (for example, bedding, foliation, etc.) is considered to represent the best way of visually differentiating it from the overlying mottled zone and underlying saprolite respectively.

The saprolite comprises bedrock that is highly weathered, but where primary rock fabrics such as bedding, foliation, etc., are still preserved. It typically comprises two subdivisions namely saprolite and saprock. Saprolite is weathered rock in which at least twenty percent and possibly all weatherable primary minerals have been either pseudomorphically replaced or dissolved to leave voids. Saprock is typically defined as rock that is partially weathered where less than twenty percent of weatherable minerals have been replaced.

Despite these subdivisions, it should be noted that the regolith profile is gradational in nature and as a consequence it is inherently difficult to subdivide. Colour and texture variations represent the best ways of subdividing and logging the regolith profile in the field.

Whilst the regolith profile is vertically predictable, it is not uncommon for subdivisions to be absent due to weathering variations or erosion and truncation. Furthermore, whilst the regolith comprises predominantly residual, in-situ material, parts of the pedolith can be transported. Evidence for displaced material include abrupt changes in the subdivisions (rather than being gradational), the presence of cobbles or boulders that are significantly less weathered than the surrounding unconsolidated material and slip surfaces.

In the Project area, parts of the Ampasibitika prospect are associated with steep slope gradients and regolith material that has been displaced from higher ground due to slope instability. Based upon the available drilling data, the thickness of the regolith in the Project area averages approximately 13.5 m, but has attained thicknesses of greater than 40 m.

X-Ray Diffraction (XRD) analysis of samples from the ferruginous zone identified iron and aluminium- rich minerals including hematite (Fe_2O_3), goethite ($FeO(OH)$) and gibbsite ($Al(OH)_3$). Clay minerals include kaolinite and illite. XRD analysis of the underlying more clay-dominant sections of the regolith profile identified lower amounts of iron and aluminium minerals and greater amounts of clay minerals including kaolinite, illite and smectite.

5.2.4 Mineralization

5.2.4.1 Overview

The Ampasindava Project includes both bedrock and regolith-hosted REE and other related rare metal mineralisation. More specifically, bedrock mineralisation is host by peralkaline rocks of the Ambohimirahavavy igneous complex and ion adsorption clay-type REE mineralisation occurs within the overlying regolith profile.

Harena Resources is focused on the exploration, delineation and ultimately exploitation of regolith-hosted REE mineralisation analogous to the ion adsorption REE mineralisation found in China. See Section 4.7.1 for a detailed description of the deposit model and characteristics. The fasibitikite dykes, skarn and other hard rock REE mineralization targets are still considered valid; however, they appear significantly less attractive at this time simply based on the tonnage, grade and conceptual mining and processing costs.

5.2.4.2 Bedrock Hosted REE Mineralisation

Alkaline igneous rocks are the main source (as progenitor or as a host) of rare earth elements and in general terms they are defined as rocks that are deficient in silicon (Si) relative to sodium (Na), potassium (K) and calcium (Ca). This means they typically contain Na- and K-bearing minerals such as the felspathoids, alkali pyroxenes and amphiboles not commonly found in other rock types (BGS, 2010). Alkaline rocks can be further classified as peralkaline if they have a higher proportion of Na and K than aluminium (Al), i.e. $Na_2O + K_2O > Al_2O_3$. Importantly, alkaline igneous rocks are typically characterised by enrichment in rare earth elements and other metals including niobium, tantalum, thorium, titanium, uranium and zirconium.

Examples of alkaline igneous REE deposits include Bokan Mountain, USA; Thor Lake, Strange Lake and Kipawa Lake in Canada; Kola Peninsula, Russia; and Ilímaussaq, Greenland (USGS, 2011).

Tectonically, alkaline igneous rocks are general associated with intra-continental rift and fault systems and can be preferentially emplaced along these structures. Mineralogically, they contain a variety of REE minerals that include REE-bearing carbonates, phosphates or fluorates, for example, allanite, apatite, bastnäsite, eudialyte, gadolinite, monazite, xenotime and zircon.

The origin of the rare earth elements is crystallisation through magmatic processes, but enrichment may also occur because of precipitation of minerals from a magmatic hydrothermal solution or redistribute of magmatic rare earth elements by the hydrothermal fluid (USGS, 2011). Alteration halos can develop around some alkaline intrusions derived from alkali-rich hydrothermal fluids.

Exploration for alkaline igneous REE deposits includes the application of geological, geochemical and geophysical methods. Because of the physical properties of several of the elements associated with this type of mineralisation, geophysical surveys methods are particularly useful for regional identification. The presence of thorium and uranium, and often the presence of potassic alteration, makes radiometric surveying particularly applicable. Radiometric methods measure the naturally emitted gamma radiation derived from three radioactive elements (potassium, uranium, and thorium) which occur in soils and rocks within the upper 0.3 m to 0.5 m of the surface. Because of the comparatively unique mineralogy of alkaline igneous rocks, geochemical sampling also provides a useful method of identification beneath areas covered by regolith.

The Ambohimirahavavy igneous complex is associated with a variety of mineralised rocks that are enriched in REEs and other rare metals. These most prevalently occur within peralkaline granitic dykes and sills, locally and historically termed fasibitikite. However, the more fractionated parts of the complex and other types of intrusions also have high contents of REE, Nb and Zr (Ganzeev and Grechishchev, 2003).

Work completed on the peralkaline granitic dykes and sills by Ganzeev and Grechishchev (2003) identified that the areas to the north and south of the Ampasibitika study area were more enriched in REE and rare metals. This was attributed to a vertical zonation of the mineralisation with the central area being much more deeply eroded than those to the north and south. It was also observed that the distribution of the mineralisation within individual intrusives was also zoned and corresponding to textural variations with higher grades associated with intrusive margins and finer grain sizes than the coarser-grained to pegmatitic interiors of some intrusives.

The main REE and rare metal minerals identified by Ganzeev and Grechishchev (2003) include chevkinite, eudialyte, monazite, pyrochlore (including a columbitised variety), thorite, and zircon. REEs, tantalum and niobium are mainly concentrated in pyrochlore. Besides pyrochlore, REE are also concentrated in zircon, eudialyte, chevkinite and monazite. REE and rare metal mineralisation results were variable with Total Rare Earth Oxide (TREO) = 0.1% to 4% (averaging 0.6%), Ta_2O_5 = 0.01% to 0.1% (averaging 0.037%), Nb_2O_5 = 0.1% to 1% (averaging 0.34%) and ZrO_2 = 0.21% to 3.84% (averaging 2.31%).

The mineralisation in the peralkaline granitic dykes and sills occurs as disseminated chevkinite, eudialyte, monazite, pyrochlore and zircon (Estrade, 2011a; 2011b). Research relating to the mineralogy of the skarns occurring at the contact between the peralkaline granitic intrusives and limestone was also completed. This established that the skarns are associated with secondary hydrothermal mineralisation comprising REE fluoro-carbonates after Na-pyroxenes (bastnäsite, synchisite, parisite and intermediate phases), titanite, pyrochlore and pseudomorphs of zircon (Ca- zirconosilicates gittinsite-zektzerite). Gangue minerals associated with the skarn mineralisation include quartz, calcite, fluorite and iron-oxides.

It was concluded that the primary mineralisation in the peralkaline granitic dykes and sills formed by crystallisation directly from magma enriched in REEs and other rare metals, and that the secondary replacement phases were transported by hydrothermal solutions (Estrade, 2012). Given the presence of fluorine-bearing minerals in the skarn assemblage, it is considered likely that the REEs and other rare metals were transported in the hydrothermal fluid by fluorine-complexing. Interaction of the fluid with the calcareous country rock caused fluorite precipitation and subsequent local decrease in REE and rare metal solubility, causing their precipitation.

In summary, the main rare earth elements and other rare metals identified in association with the Ambohimirahavavy igneous complex to date include:

- **Chevkinite** $(Ca,Ce,Th)_4(Fe,Mn)_2(Ti,Fe)_3Si_4O_{22}$
- **Baddeleyite** ZrO_2
- **Bastnäsite** $(Ce,La)(F/CO_3)$
- **Columbite** $FeNb_2O_6$
- **Eudialyte** $Na_{15}Ca_6(Fe,Mn)_3Zr_3SiO$
- **Gagarinite** $NaCaY(F,Cl)_6$
- **Microlite** $(Ca,Na)_2Ta_2O_6(O,OH,F)$
- **Monazite** $(Ce,La,Nd,Th)PO_4$
- **Parisite** $Ca(Ce,La)_2(CO_3)_3F_2$

Most of the minerals are fine-grained, less than 1 mm, with only subordinate coarse-grained phases as zirconium-hafnium-REE, eudialyte and zircon. Niobium and tantalum mineralisation mainly occur in association with pyrochlore and columbite.

The REE and rare metal mineralisation is associated with an elevated radiometric response, with gamma-activity of 80 µr/h to 1,450 µr/h (averaging 300 µr/h), relative to an estimated background of 25 µr/h to 40 µr/h. A direct correlation between the content in niobium pentoxide and the gamma radioactivity has been established (correlation coefficient = + 0.69). This geophysical characteristic means that radiometric methods are well suited for further exploration in the area. However, even though the mineralisation is associated with anomalous radioactivity, the overall uranium and thorium content of the in-situ bedrock (based upon the available geochemical results) is relatively low and averages 12 ppm U_3O_8 and 57 ppm ThO_2. These concentrations are not considered to pose any environmental or anthropogenic risks.

5.2.4.3 Regolith Hosted REE Mineralisation

During 2009, it was recognised that the regolith overlying the Ambohimirahavavy igneous complex was also mineralised with REEs and that this material may be similar to the ion adsorption clay-type REE mineralisation exploited in China. Subsequent independent testwork has confirmed the presence of REEs that are ionically-adsorbed onto clay minerals and that are amenable to leaching and the recovery of REEs.

The Chinese ion adsorption clay-type REE mineralisation was first identified in the late 1960's (Chi and Tian, 2008). There are reportedly more than 200 deposits with 90% of them occurring in the southern provinces, principally Jiangxi, Hunan, Guang Dong, Guang Xi and Fujian (Bao and Zhao, 2008). The reason for this apparent geographical control is the climatic conditions required to weather the bedrock to form the regolith host material (generally a sub-tropical environment south of 28°N with warm, humid conditions and rainfall exceeding 1500 mm per year).

Ion adsorption REE mineralisation can be summarised as REEs that are mainly adsorbed onto the surfaces of clay minerals in the form of hydrated ions or hydroxyl-hydrated ions. These ions are derived from bedrock-hosted REE mineralisation that has been weathered resulting in the liberation and mobilisation of the REEs.

Most of the exploited Chinese deposits are formed from the weathering of highly evolved Mesozoic granites, but some have also developed from the weathering of other rock types including volcanics and lamprophyre. The main REE bearing accessory minerals in the Chinese source rocks are allanite, bastnäsite, doverite, gadolinite, monazite, parisite and xenotime. Accessory minerals contain the majority of the REEs (more than 70%) with the remaining percentage occurring within rock-forming minerals (Bao and Zhao, 2008).

The regolith material hosting the Chinese deposits typically ranges in thickness between 8 m and 10 m thick (Chi and Tian, 2008). In the available literature, the regolith profile is simplistically described as consisting of an upper soil zone (2 m to 5 m thick), an underlying weathered zone (5 m to 30 m thick) and lower sub-weathered zone (5 m to 8 m thick). Further subdivisions are noted as including pedolith, ferruginous, mottled clay and rock fragment zones (Zuoping and Chuanxian, 1996).

The two fundamental controls on the formation of ion adsorption REE mineralisation are the availability of an REE-enriched source rock and in-situ sub-tropical weathering conditions that enable the liberation and mobilisation of the REEs and their preferential adsorption onto the surfaces of clay minerals. In the Chinese deposits, 60% to 90% of the REEs are adsorbed onto kaolinite with other clay minerals including montmorillonite and halloysite (Chi and Tian, 2008). Approximately 10% of the REEs occur as mineral phases in the form of bastnäsite, monazite and xenotime. REE mobilisation and accumulation in the regolith profile appears to be controlled by the mineralogy of the REE- enriched source rocks, specifically the type, abundance, distribution and stability of the primary REE minerals during weathering.

REE fractionation is directly proportional to the intensity of weathering and REE content typically increases with depth and then decreases approaching the un-weathered bedrock. The REE content of the regolith is generally two to four times greater than the underlying bedrock, but has been reported as being up to seven times greater (Zuoping and Chuanxian, 1996).

Exploration

There is very little documented information on the methods used to explore for ion adsorption-REE mineralisation in China. The principal methods appear to be visual identification of mineralised material and recognition of favourable geomorphological features. Given that the REEs are not discernible with the naked eye, visual identification of mineralised material involves the colour of the regolith. For example, yellow, pale-red or white coloured material (Chi and Tian, 2008). Geomorphologically, favourable accumulations of regolith are best developed where the topography is gentle and denudation rates are low. REE enrichment is also apparently greater on ridges and elevated features than in gullies.

Deposit Size and Economics

Ion adsorption REE mineralisation is characteristically low-grade. The Chinese deposits generally contain between 0.05% and 0.35% TREO, but there is considerable variability in grade even within the same deposit. Grades of greater than 0.05% TREO in the presence of sufficient volumes are typically considered to be economic (Bao and Zhao, 2008). Exploited grades as low as 0.01% / 100 ppm TREO are also reported (Orris and Grauch, 2002).

Individual deposits are relatively small and typically range in size from 1,500 to 12,000 t TREO (Orris and Grauch, 2002). Annual production is reportedly approximately 10,000 t TREO per year (Bao and Zhao, 2008) and proven reserves in the order of 1.48 Mt TREO (Chi and Tian, 2008).

Ion adsorption REE deposits are economically important because they contain a significant proportion of rarer and more valuable HREEs compared with other types of REE mineralisation. For example, bedrock-hosted deposits such as Mountain Pass in the USA and Bayan Obu in China contain a much higher proportion of LREEs. Ion adsorption REE deposits are also considered favourable because they are associated with low levels of radioactivity and are simpler and less expensive to exploit compared with their bedrock counterparts. However, in-situ leaching has also been known to cause significant environmental damage, particularly by illegal miners in southern China (Chinafolio 2014).

Processing

Ion adsorption REE deposits can only be exploited chemically. Exploitation of the Chinese deposits involves batch, heap and in-situ leaching of the ionic material using either sodium chloride or ammonium sulphate with recoveries reportedly ranging from 40% to 99% (Orris and Grauch, 2002).

Summary

The regolith material in the Ampasindava Project area has many similarities to the material in southern China: both developed in a sub-tropical environment with warm, humid conditions and significant rainfall; they have comparable thicknesses; both have variable but similar REE grades that generally increase with depth and are associated with an increased proportion of HREE; both contain “ionic clays” that adsorb REEs and, as with the Chinese examples, the Ampasindava Project has, from preliminary testwork shown that it may be amenable to leaching using comparatively inert solutions in order to recover the REEs; and both are associated with only low levels of radioactivity.

It is difficult to make comparisons between the source rocks and the actual regolith profiles due to the lack of available data. Similarly, it is difficult to compare the mineralogy of the mineralisation in the regolith profile due to insufficient data.

5.3 Previous Exploration

5.3.1 Colonial Exploration

Colonial-era exploration activities completed by French Geologists first noted ‘peculiar’ granitic intrusive rocks near to the village of Ampasibitika in the late 19th century. This was followed by mineralogical studies of the rocks, uniquely named fasibitikite, and the documented description of niobium-tantalum-zirconium mineralisation (Lacroix, 1922).

Between the 1920’s and the 1970’s, work in the area mainly consisted of academic research. However, during this time, the Ampasindava Peninsula was also geologically mapped by the Governmental Service Géologique at a scale of 1:200,000 (sheet PQRS34-35 Anorotsangana- Ambanja) and published in 1958.

5.3.2 Soviet Exploration

Between 1988 and 1991, a Russian-funded exploration program termed the Soviet Geological Mission was completed in conjunction with the Malagasy Office Militaire National pour les Industries Stratégiques (OMNIS).

Russian Geologists undertook a program that included systematic stream sediment and outcrop sampling, ground radiometric surveying and pitting. They also completed the first detailed mapping of the mineralised intrusives along a 2 km stretch of coastline in the vicinity of Ampasibitika village.

The program speculated that radiometric survey results over visible mineralised intrusions could be extended along strike under the regolith cover and through areas of poor outcrop. However, it is now understood that, because radioactive emissions can only be detected from material at or very near the surface, it was most likely recording the radiometric response of relict uranium and thorium minerals present in the regolith rather than the actual bedrock mineralisation.

The pitting programme involved the excavation of a series of shallow pits on a 100 m by 400 m grid that aimed to expose the extent and nature of mineralisation at depth. In total, eleven pits were excavated (totalling 55 m) with all but one reaching fresh rock. Their typical dimensions were 1 m by 1.35 m with depths that varied from 2.75 m to 6.75 m. Over the course of the program, the mapping component extended out from the main study area and covered an area of 10 km^2 at a scale of 1:50,000. Preliminary metallurgical testwork was carried out on pit samples to determine possible concentrating techniques for the observed Uranium mineralisation. The results demonstrated some success with gravity and magnetic techniques.

A list of the work completed as part of the Soviet Geological Mission is provided in Table 5-1. This period was followed by an episode of political instability in Madagascar and during the 1990's and early 2000's no exploration work was conducted in this area.

Table 5-1 Exploration Completed as part of the Soviet Geological Mission (after OMNIS-SM, 1989; 1990; 1992b; 1992c; 1992d; 1992e)

Type of geological work	*Unit of measure*	*Planned*	*Actual*
Geological prospecting at 1:10,000 scale			
Radiometric surveys	Line km	18	18
Excavation of pits	Line m	55	55
Geochemical sampling around aureoles of secondary dispersion	Sample	750	748
Geochemical sampling of outcrops	Sample	0	55
Geochemical channel-sampling of pits	Sample	55	55
Geochemical channel-sampling of outcrops	Sample	0	15
Line cutting	Line km	17	13.4
Pegging profiles and baselines at 25 m intervals	Line km	20.4	21
Research at 1:50,000 scale			
Research traverse	Line km	25	25
Geochemical sampling of outcrops	Sample	0	22
Litho-geochemical investigation following the traces of dispersion	Sample	100	122

5.3.3 Contemporary Exploration and Results

5.3.3.1 Stream and beach sediment sampling

In 2008, Fugro Consult GmbH (Fugro) was commissioned by the then owners of the Ampasindava project area, Zebu Metals Ltd (Zebu) to undertake a week-long reconnaissance field program. As part of this program Fugro collected five beach sediment samples along the eastern edge of the project area. However, no major accumulations of heavy minerals of interest were identified.

5.3.3.2 Bulk Sampling

In 2008, as part of the Zebu-commissioned program, Fugro confirmed the widespread occurrence of mineralised peralkaline granitic intrusives in the vicinity of Ampasibitika village and collected two mini bulk samples weighing 60 kg and 80 kg for geochemical analysis. The aim of this sampling was to study the mineral ratios and overall grades of the mineralisation. The results of the bulk sampling are summarised in Table 5-2.

Table 5-2 Fugro 'Mini Bulk Sample' Results (Fugro, 2008)

Sample No.	*TREO+ Y2O3* (ppm)	*Nb2O5* (ppm)	**Ta2O5** (ppm)	**Sn** (ppm)	**U** (ppm)	*ZrSiO4* (%)
476323	4427	1932	165	137	115	5.37
476324	3332	4107	336	200	207	2.8

5.3.3.3 Airborne Geophysical Surveys

In 2008, Fugro Airborne Surveys of South Africa completed a helicopter-borne magnetic and radiometric survey. Between the 4th and 8th of July a total of 2,936 line kilometres were flown at a line spacing of 100 m and a bearing of 045°. Tie lines were flown every 1000 m on a bearing of 135°. In total, the survey covered an area of 244.4 km^2. The full survey specifications are detailed in a separate report by Fugro Consult (2008).

In 2009, a geological interpretation of the magnetic and radiometric data was complete by Mr. K. P. Knupp of Earthmaps Consulting. The pertinent mapping and interpretation are presented in Section 5.2 and complete details are provided in a separate report (Earthmaps Consulting, 2009). Examples of radiometric and magnetic data images are provided in Figure 5-6 and Figure 5-7 respectively.

Figure 5-6 Map Showing the Radiometric Ternary Imagery for the Ampasindava Project (after Earthmaps Consulting, 2009)

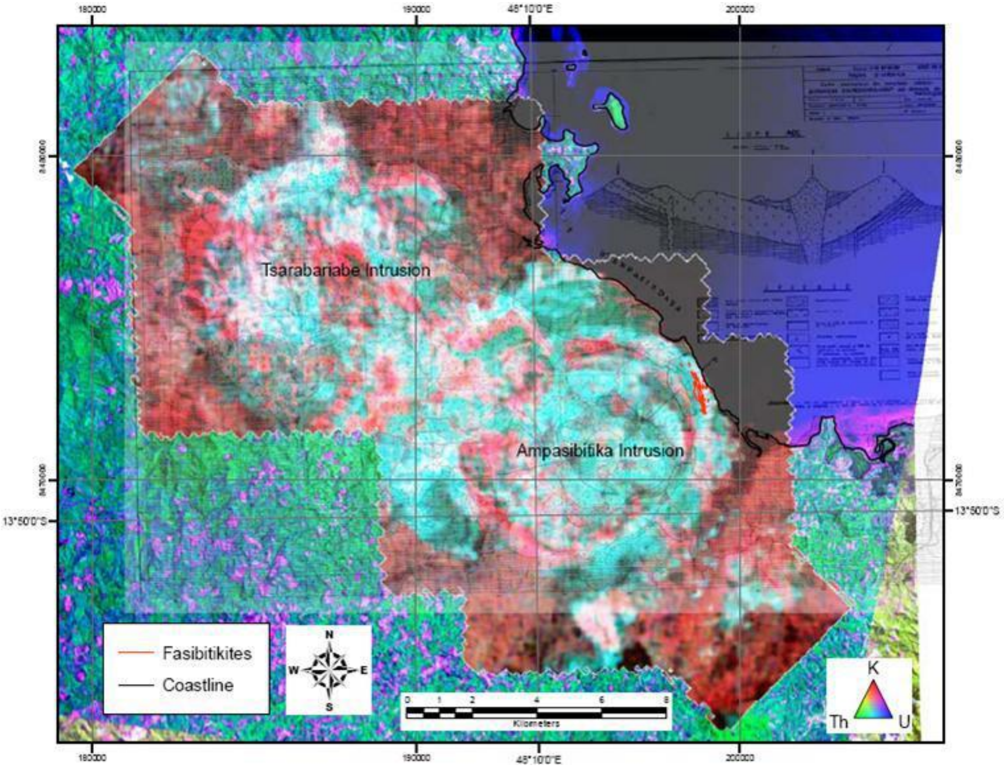
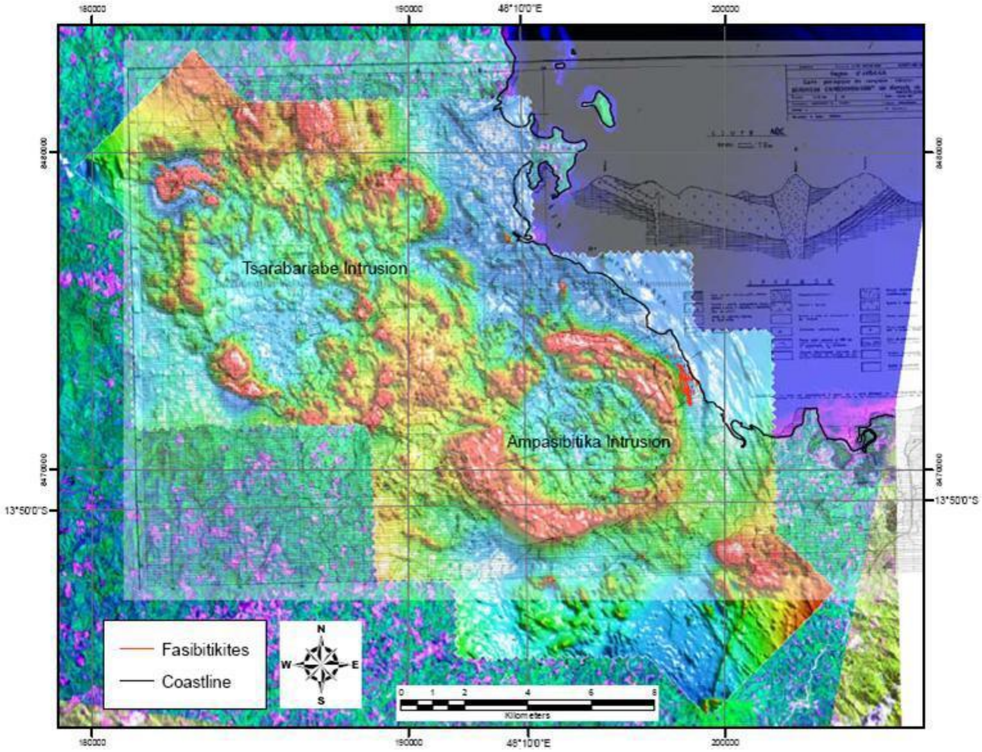


Figure 5-7 Map Showing the Magnetic Imagery for the Ampasindava Project (after Earthmaps Consulting, 2009)



5.3.3.4 Outcrop Sampling

Tantalus collected and analysed a total of 284 outcrop samples from within the project area. These were predominantly peralkaline intrusive rocks collected from areas associated with radiometric anomalies. The locations of the outcrop samples are shown in Figure 5-8 and are summarised by prospect in Table 5-3.

The results have confirmed the presence of bedrock-hosted REE mineralisation in known areas, as well as identifying new areas of mineralisation. The highest grade samples are associated with peralkaline granitic rocks derived from the Ampasibitika prospect (up to 22,408 ppm / 2.24% TREO). Of note is that none of the Caldera prospect outcrop samples are peralkaline granite. The vast majority are volcanic breccia that is also evidently enriched in REEs (up to 8,201 ppm / 0.82% TREO).

Figure 5-8 Map Showing the Locations of the Outcrop and Soil Samples

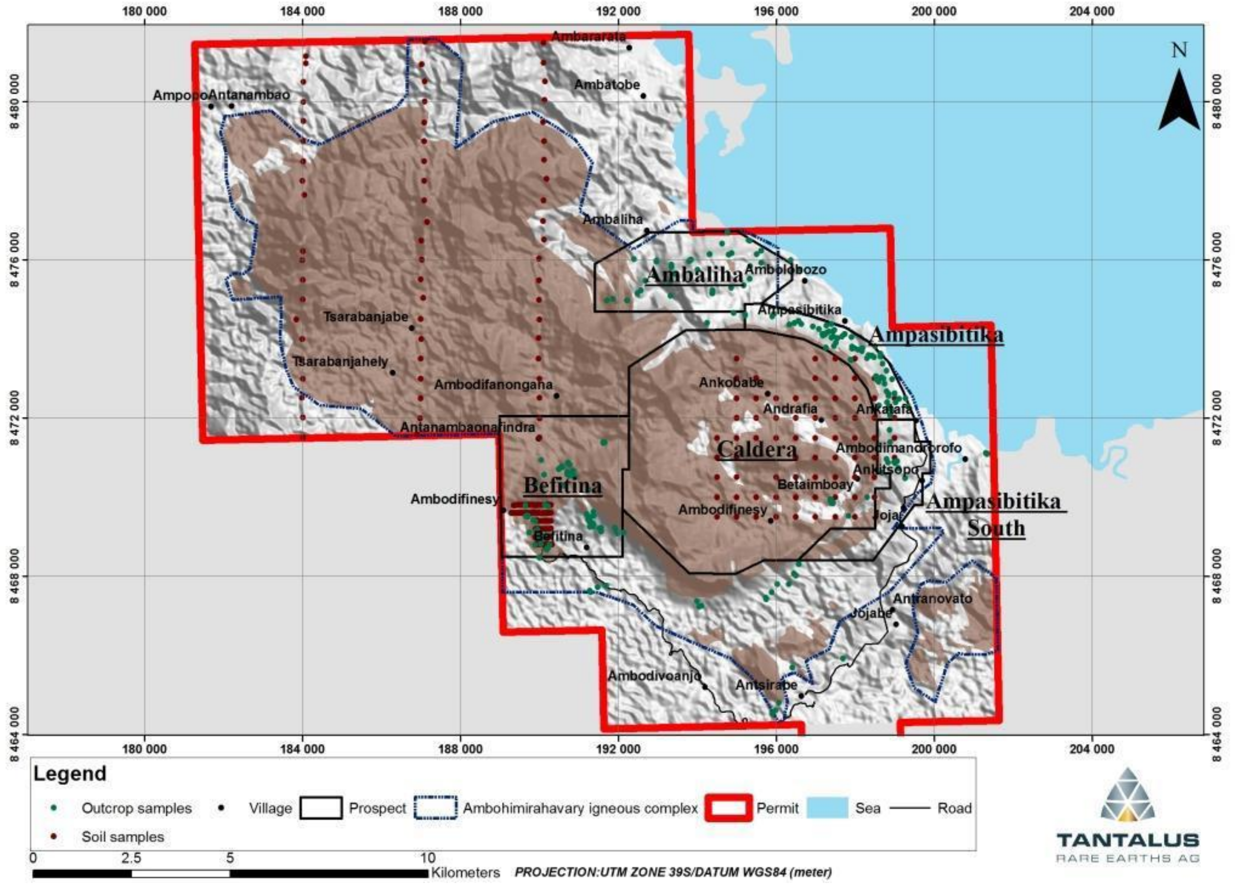


Table 5-3 Summary of the Tantalus Outcrop Sample Results (Gilbertson, 2013)

PROSPECT	*STATS*	*TREO (ppm)*	*HREO (%)*	*Nb_2O_5 (ppm)*	*Ta_2O_5 (ppm)*	*ZrO_2 (ppm)*	*HfO_2 (ppm)*	*Ga (ppm)*
All prospects	MIN	45	5	6	0	39	1	2
	MAX	22,408	69	10,544	891	58,760	1,480	78
	MEAN	2,108	26	949	70	5,442	116	36
	MEDIAN	863	23	386	23	1,530	33	37
	nSamples	284						
Ampasibitika	MIN	45	5	6	0	39	1	2
	MAX	22,408	47	10,544	891	58,760	1,480	69
	MEAN	3,031	24	1,698	125	10,512	224	38
	MEDIAN	1,134	22	484	30	1,905	43	41
	nSamples	91						
Ambaliha	MIN	100	12	9	1	174	4	6
	MAX	7,582	69	2,768	200	12,387	281	49
	MEAN	1,842	28	770	51	3,753	82	34
	MEDIAN	1,096	24	419	29	1,986	46	36
	nSamples	43						
Befitina	MIN	161	14	19	1	204	4	16
	MAX	13,576	65	3,419	368	22,288	479	57
	MEAN	1,825	27	649	52	3,634	78	36
	MEDIAN	781	24	386	24	1,736	37	36
	nSamples	97						
Caldera	MIN	257	17	131	8	507	13	22
	MAX	8,201	65	538	26	1,594	33	40
	MEAN	1,824	31	178	10	631	16	26
	MEDIAN	988	30	153	10	558	15	26
	nSamples	19						
Ampasibitika South	MIN	365	7	204	11	686	12	26
	MAX	10,684	30	3,820	244	19,181	320	73
	MEAN	1,650	22	789	46	3,301	62	49
	MEDIAN	676	22	452	24	1,604	34	44
	nSamples	10						
Other	MIN	223	12	71	4	207	4	16
	MAX	2,349	28	845	48	4,336	88	78
	MEAN	642	21	325	18	1,253	26	34
	MEDIAN	476	22	278	15	896	18	32
	nSamples	24						

5.3.3.5 Soil Sampling

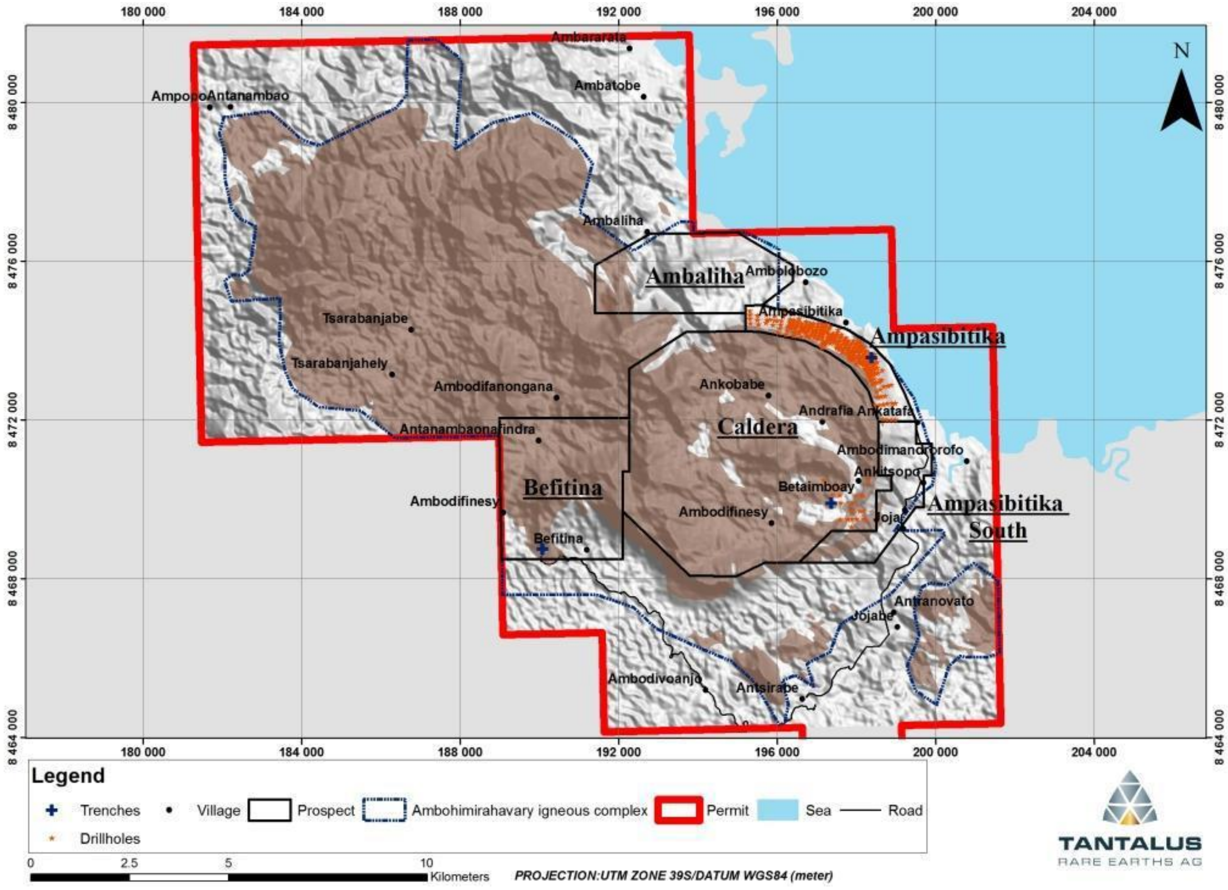
Tantalus completed soil sampling in several parts of the Ampasindava project area, namely parts of the Befitina, Ampasibitika and Caldera prospects and several lines across the northwest of the Property. This exploration method was utilised due to the lack of outcrop in these areas. These samples have now been superseded by the pitting samples which currently cover most of the Property.

5.3.3.6 Trenching

A total of five trenches have been excavated in the Project area, one in the Ampasibitika prospect and two in each of the Befitina and Caldera prospects Figure 9-2

The single trench in the Ampasibitika prospect was excavated and sampled in 2008 by Fugro as part of the Zebu-commissioned programme (Fugro Consult, 2008). The 30 m long, east-west orientated trench (TANT1) was manually excavated above a weathered peralkaline granite intrusive and surrounding regolith. The depth of the trench averaged 0.7 m but it did not reach bedrock over its entire length. A total of 16 contiguous channel samples were collected from the trench (each corresponding to a length of approximately 2 m). The best TREO, Nb, Ta and Zr grades corresponded to samples that included bedrock material. These returned average grades of just over 2,000 ppm / 0.2 % TREO. Regolith samples comprising clayey soil contained an average of 1,000 ppm / 0.1 % TREO.

Figure 5-9 Map Showing the Locations of the Trenches and Drillholes



In the Befitina prospect, two 100 m long trenches were manually excavated and sampled (TANT2 and TANT3). The two trenches were excavated perpendicular to one other, crossing at about their mid points. The depth of the trenches was 3 m, but did not intersect bedrock. Twenty contiguous horizontal channel samples were collected from trench TANT2, and 22 samples were collected from trench TANT3 (each corresponding to a length of approximately 5 m). The sample results for trenches TANT2 and TANT3 are summarised in Table 5-4. In the Caldera prospect, two 100 m long trenches were manually excavated and sampled (TANT4 and TANT5). The two trenches were excavated perpendicular to one another and attained depths of greater than 4 m. Trench TANT4 intersected predominantly regolith (pedolith and saprolite), but also bedrock in a few places. Trench TANT5 intersected only saprolite. In total 100 horizontal channel samples were collected from each trench (each corresponding to a length of approximately 1 m). The sample results for trenches TANT4 and TANT5 are summarised in Table 5-4.

Table 5-4 Summary of the Tantalus Trench Sample Results (Gilbertson, 2013)

PROSPECT	*STATS*	*TREO (ppm)*	*HREO (%)*	*Nb_2O_5 (ppm)*	*Ta_2O_5 (ppm)*	*ZrO_2 (ppm)*	*HfO_2 (ppm)*	*Ga (ppm)*
TANT2	MIN	899	16	383	28	1,932	50	48
(Befitina)	MAX	3,663	39	1,753	144	8,456	213	73
	MEAN	1,612	28	1,033	79	4,820	121	59
	MEDIAN	1,328	29	974	71	4,593	116	57
	nSamples	20						
TANT3	MIN	661	13	192	12	1,299	30	49
(Befitina)	MAX	2,666	38	1,788	112	6,889	157	71
	MEAN	1,397	26	950	63	4,403	101	61
	MEDIAN	1,272	27	1,023	70	4,944	116	59
	nSamples	22						
TANT4	MIN	467	11	131	9	401	12	30
(Caldera)	MAX	2,640	35	439	24	1,526	31	66
	MEAN	1,107	22	232	14	812	21	40
	MEDIAN	982	21	228	14	812	21	40
	nSamples	100						
TANT5	MIN	470	16	183	11	655	17	32
(Caldera)	MAX	4,589	31	383	21	1,322	30	55
	MEAN	1,264	23	232	14	819	23	41
	MEDIAN	1,108	22	234	14	805	22	40
	nSamples	100						

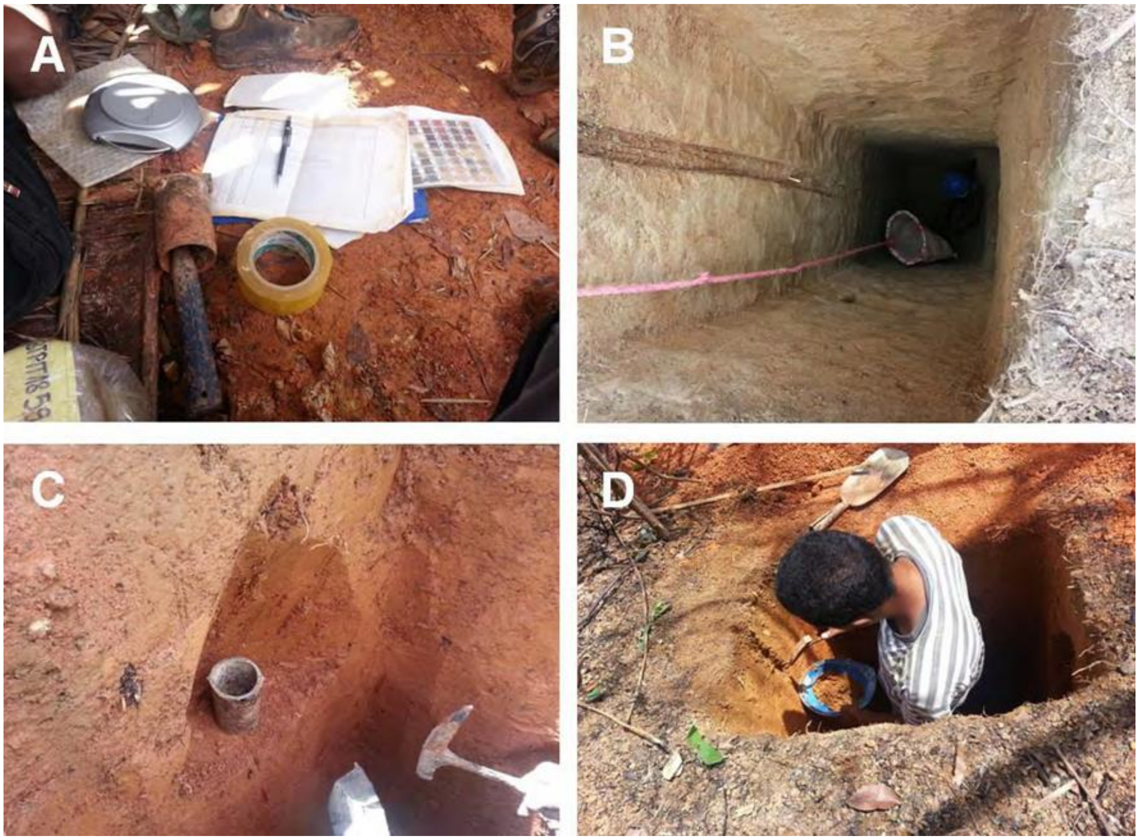
The summary trench sample results show an interesting trend. Whilst the average TREO (ppm) results for the Befitina and Caldera prospect trenches are similar, those for the other rare metals are evidently dissimilar. That is the Nb, Ta, Zr and Hf results from the Befitina prospect are significantly higher than those from the Calder prospect. This is an interesting trend that can be explained by the differences in lithological setting, with the Befitina prospect comprising sedimentary rocks that contain mineralised intrusives that host certain rare metals and the Caldera prospect that is predominantly volcanic breccia. From an economic perspective, it suggests that the regolith material is similarly enriched in REEs. This is significant as it substantiates the prospectivity of areas underlain by volcanic breccia in line with those underlain by mineralised intrusive rocks. However, it should be noted that this is too small a dataset on which to make definitive conclusions.

5.3.3.7 Pitting

Tantalus manually excavated a total of 4,474 pits for the purposes of assessing regolith- hosted REE mineralisation. The pits were excavated manually in 2011 and 2013. They are vertical pits typically 1 m by 1 m with a vertical depth of up to 10m, with an average of 5.68m. These have now been excavated over the six prospects with spacing ranging from 50 m to 250 m (Figure 4-1Figure 5-11).

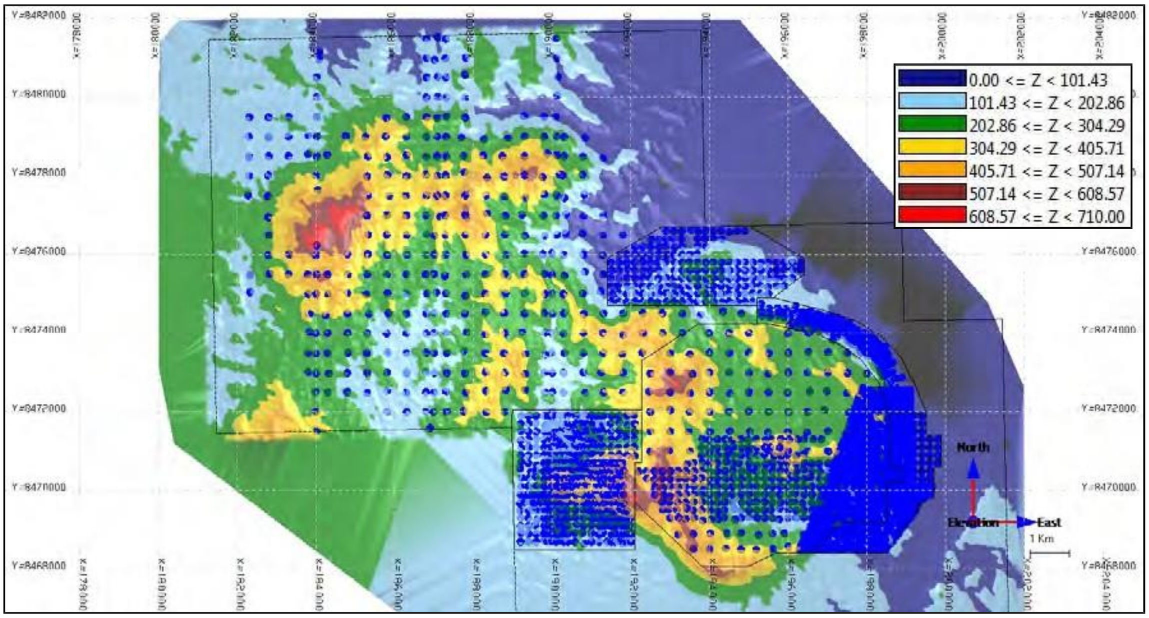
Ideally the pits were excavated to bedrock. However, for safety reasons the pits were not excavated deeper than 10 m. It took, on average, 4 days to manually excavate each pit. Once a pit was excavated, the sampling methodology involved marking out the samples on the same wall of each pit at 1.0 m intervals (0.5m lengths are found in the 2011 data). Samples were collected from the lowermost interval first to minimise contamination. Collection involved using the pointed end of a rock pick or machete to create a continuous vertical channel with the displaced material collected in a bucket or a polythene sample bag with an average sample weight of 1.8 kg. All of the pits were back- filled as soon as geological observations, density measurements, moisture readings and sampling were completed. A photograph of a typical exploration pit is shown in Figure 5-10 and the summary statistics for the pits excavated to date are provided in Table 5-5.

Figure 5-10 Photos of Test Pits



Note: A. Geologists field tools and protocols; B. Ongoing pit showing the typical sampling trench and branches inserted to track the depth within pits. C. Ledge cut into the side of the pit used to collect the sample for density measurements. D. Geologist sampling the first meter within a pit.

Figure 5-11 Topographic Image Showing the Distribution of Pits within the Project



Note: Areas that are completely covered in points are covered by a grid approaching 50m. The most sparsely drilled/pitted area have a grid spacing of around 250m. Hotter colours indicate higher elevation.

Table 5-5 Summary of Analyses from the Tantalus Pitting Campaigns

Layer*	Statistic	TREO (ppm)	TREO noCe (ppm)	LREO (ppm)	HREO (ppm)	Nb_2O_5 (ppm)	Ta_2O_5 (ppm)	ThO_2 (ppm)	U_3O_8 (ppm)
PED	**Number of samples**	13,926							
	Min	28	17	26	2	1	0	1	0
	Max	14,995	8,001	14,252	3,236	3,577	1,186	1,138	863
	Mean	786	445	643	143	232	13	54	10
	Median	635	297	521	114	193	11	47	9
SAP	**Number of samples**	16,133							
	Min	33	18	29	4	1	0	0	0
	Max	39,098	38,504	31,183	7,915	5,469	1,399	4,300	539
	Mean	1,003	720	805	198	188	11	46	9
	Median	729	451	586	136	139	8	38	7

* Layers are as modeled, not actual lithology description TREO = LREO+HREO TREO noCe = TREO-Ce2O3
HREO = Y2O3+Eu2O3+Gd2O3+Tb2O3+Dy2O3+Ho2O3+Er2O3+Tm2O3+Yb2O3+Lu2O3
LREO = La2O3+Ce2O3+Pr2O3+Nd2O3+Sm2O3

5.3.3.8 Window Sampling

Tantalus drilled a total of 47 window sampling holes using a Geotools Wacker BH23 unit (Figure 5-12). The majority of the window sampling holes (44) were drilled in the Caldera prospect with rest in the Ampasibitika South prospect. Their purpose was to assess the suitability of the technique as a faster and safer accompaniment to pitting.

The window sampling program resulted in the drilling of approximately 354 m of regolith material. Hole lengths range from 1.5 m to 11.0 m and the average length was 7.5 m. On average, one window sampling hole can be completed per day. The window samplers were not used due to technical issues, but their use is recommended due to the potential to push holes to bedrock (past 10m) and also to reduce the safety risk associated with the pit sampling. The sampling method appears to respect international standards with minimal contamination given careful application of protocols.

Figure 5-12 Window Sampler Demonstration

Note: A. Window sampler in action. B. Sample tube with fall back material from previous meter (in white box). C. Contaminated material removed. D. Cleaning of sample tube prior to next run.

5.3.3.9 Core Drilling

The initial Tantalus exploration strategy specifically focussed on exploration for bedrock-hosted REE mineralisation and in 2010 the decision was made to drill the radiometric anomaly of the Ampasibitika prospect. Between July 2010 and October 2011, E Global Drilling Corp (a subsidiary of Energold Drilling Corp) was contracted to complete the drilling. The drilling involved the use of three rigs: two Energold EGD II‟s, and a Versadrill Kmb.4km rig. The Energold rigs were man-portable and the Versadrill was adapted to become man-portable. Local teams were hired to work as off-siders as well as for rig moves. A photograph showing one of the drill-rigs in operation is provided in Figure 5-13.

A total of 277 holes were drilled in the project area, equating to 20,084.6 m of NW (7.62 cm diameter), NTW (5.61 cm diameter), and BTW (4.17 cm diameter) core. Drillhole lengths ranged from 42.2 m to 130.0 m and the average drillhole length was 72.5 m. The average daily metreage rate per drillhole was 26.4 m. The locations of the drillholes are shown in Figure 5-9.

Figure 5-13 Photograph of the Versadrill Kmb.4km Drill Rig in Operation (Gilbertson, 2013)

The drilling program encompassed a 5 km by 4 km section of the eastern and north-eastern flank of the Ambohimirahavavy igneous complex. Holes were ultimately drilled on 100 m to 200 m spaced fences typically comprising eight drillholes spaced at 50 m intervals. Drilling commenced in the south of the prospect on 400 m spaced fences and proceeded northwards across the radiometric anomaly. Infill drilling was subsequently completed. The majority of the holes were drilled at an angle of -70° with azimuths to the east and northeast, but also included holes drilled at -45° and vertically. All of the drillholes were cased through the regolith and bedrock core recovery was consistently good and typically greater than 90%.

An additional 20 holes were also drilled in the south-eastern part of the Caldera prospect, equating to 2004.07 m of NW (7.62 cm diameter) and NTW (5.61 cm diameter) core. Drillhole lengths ranged from 100.0 m to 100.8 m and the average daily metreage rate per drillhole was 50.3 m. The locations of the drillholes are shown in Figure 5-9. The purpose of the drilling programme was to test for the presence of bedrock-hosted REE mineralisation within the volcanic breccia occurring within the caldera. All of the holes were drilled vertically.

5.3.3.10 Exploration Conclusions

The bulk of data used for the estimation of Mineral Resources is derived from the manual excavation of pits. Careful observation of the method during the site visit in 2013 has confirmed that the protocols used are appropriate and the results of the sampling can be used for the resource estimation.

5.4 Sample Preparation, Analyses, and Data Verification

5.4.1 Overview

The following section summarises the sample preparation, analyses, and security measures implemented for the Project and details the data verification measures taken by SGS to ensure that the Project data was suitable for the estimation of Mineral Resources. Refer to Desharnais et al, 2016 for full details and QAQC charts.

5.4.2 Sample Preparation

The vast majority of samples were prepared at the Tantalus sample preparation facility in Ambanja. The facility is well run and should provide appropriate sampling preparation if the protocols are followed. Sample preparation procedures are described as follows.

5.4.2.1 Drill Core Samples - Bedrock Intersections

1. The core is split in half using a hydraulic splitter (core sawing is reportedly not possible due to frequent mains power outages), and half is returned to the core box;
2. The half to be sampled undergoes systematic density measurement using the immersion in water method;
3. The samples are then crushed to minus 2 mm using a Fritsch Industries RoHS 2002/86/EG electric jaw crusher. After each sample, blank material (locally sourced granitic material) is crushed and the equipment is cleaned with compressed air and a vacuum cleaner in order to minimise sample contamination;
4. The crushed samples are then split twice using a Humboldt H-3962 riffle splitter in order to produce a quarter of the sample. Of this homogenised material, 250 g to 350 g is collected using a plastic scoop and bagged for analysis. Sample numbers are written onto the polythene sample bags with permanent marker pen and an aluminium tag inscribed with the sample number is also placed into the bag.

5.4.2.2 Drill Core Samples - Regolith Intersections

1. Due to its consistency, the core is split in half using a geological hammer;
2. The samples are then weighed (inclusive of moisture) and emptied into stainless steel bowls in preparation for drying;
3. The samples are then dried in a gas oven at a temperature of 135°C for four to eight hours, depending on the moisture content of the samples;
4. Once dried, the samples are re-weighed and the weight recorded;
5. If the dried samples are observed to contain any rock fragments, they are crushed to minus 2 mm using a Fritsch Industries RoHS 2002/86/EG electric jaw crusher. After each sample, blank material is crushed and the equipment is cleaned with compressed air and a vacuum cleaner in order to minimise contamination;
6. If the dried samples contain no rock fragments, they are manually pulverised in the stainless steel bowls using a large wooden pestle;
7. The crushed samples are then split twice using a Humboldt H-3962 riffle splitter in order to produce a quarter of the sample. Of this homogenised material, 250 g to 350 g is collected using a plastic scoop and bagged for analysis. Sample numbers are written onto the polythene sample bags with permanent marker pen and an aluminium tag inscribed with the sample number is also placed into the bag.

The remaining coarse reject material is retained and stored at the sample preparation facility. The drill core is stored in a dedicated warehouse in Ambanja.

5.4.2.3 Pit and Window Samples

1. The samples are weighed (inclusive of moisture) and emptied into stainless steel bowls in preparation for drying;
2. The samples are then dried in a gas oven at a temperature of 135°C for four to eight hours, depending on the moisture content of the samples;
3. Once dried, the samples are re-weighed and the weight recorded;
4. If the dried samples are observed to contain any rock fragments, they are crushed to minus 2 mm using a Fritsch Industries RoHS 2002/86/EG electric jaw crusher;
5. If the dried samples contain no rock fragments, they are manually pulverised in the stainless steel bowls using a large wooden pestle;
6. The crushed samples are then split twice using a Humboldt H-3962 riffle splitter in order to produce a quarter of the sample. Of this homogenised material, 250 g to 350 g is collected using a plastic scoop and bagged for analysis. Sample numbers are written onto the polythene sample bags with permanent marker pen and an aluminium tag inscribed with the sample number is also placed into the bag.

5.4.3 Analytical Method

At ALS Chemex in Vancouver the samples were subject to 38-element fusion Induced Coupled Plasma Mass Spectrometry (ICP-MS) analysis (ALS code ME-MS81). This involves the addition of 0.2 g of prepared sample to 0.9 g of lithium metaborate flux, mixing and fusion in a furnace at 1000°C. The resulting melt is then cooled and digested in 100 mL of 4% nitric acid (HNO3) and 2% hydrochloric acid (HCl) solution and analysed using ICP-MS.

At the SGS laboratory in Booyens South Africa the samples were prepared analysed with the GO IMS91B package (sodium peroxide fusion with ICP-MS finish). Sodium peroxide is a strongly oxidizing flux that is basic, not acidic in nature. It renders most refractory minerals soluble. Because the fusion temperature is lower than that of lithium metaborate fusions, the hydride elements are not volatised.

Both of the ALS Chemex laboratories and the SGS laboratory are ISO accredited. The analytical methods used are according to industry standards and data provided is appropriate for use in the resource estimation.

5.4.4 Quality Assurance and Quality Control (QAQC)

Quality Assurance and Quality Control (QAQC) procedures used for the Project to validate the sample results included the routine insertion of one blank, one standard, and one duplicate sample within every 35 samples collected. This relates to an insertion rate of approximately 8 %, with one control samples in every 12 samples sent to the ALS Chemex laboratories in 2011 and SGS South Africa laboratory in 2013.

5.4.4.1 Standards

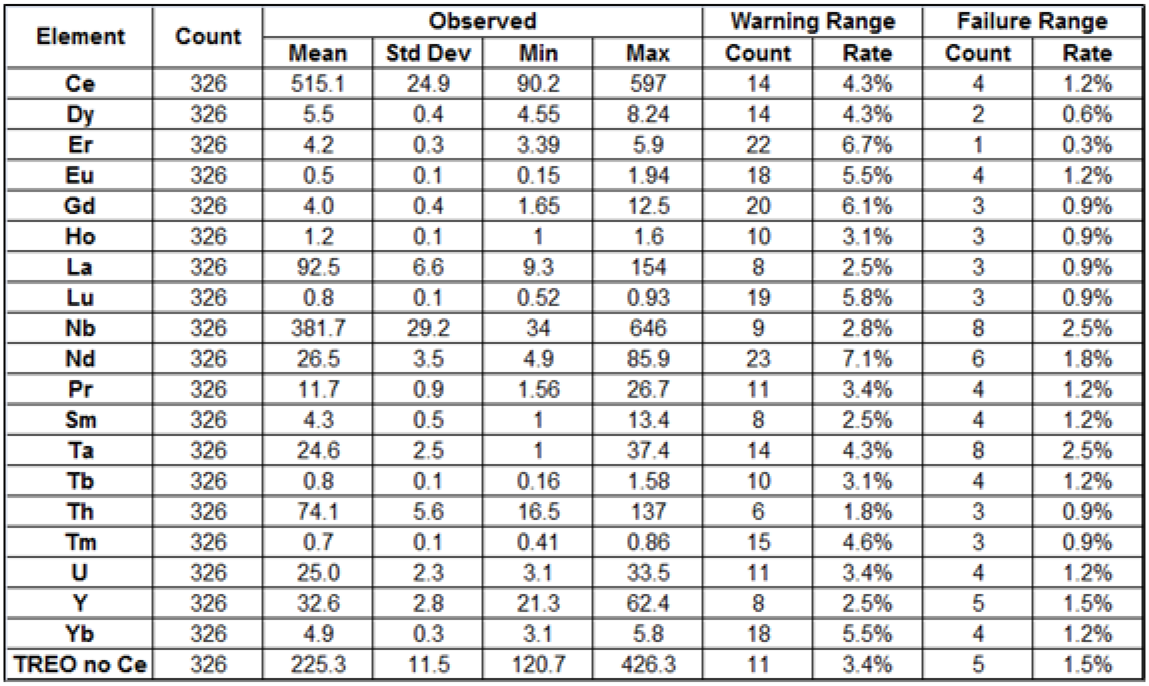
Standard material is inserted into the sample stream to test the assaying accuracy of the laboratory. To date three separate standards have been produced by Tantalus to control the accuracy of grade of the REE. These standards were sampled from the Project and have sensible grade values; unfortunately none of these standards were validated through a round robin and statistical evaluation. Two new standards were prepared by a commercial lab and sent for round robin testing; these new standards will be ready for the next exploration season.

In order to gain a level of confidence in the standards test results, SGS Geostat analysed the resulting data, for various errors. Human error with such large data input can be a common issue. Graphs depicting the standard deviation were made from the original data and outliers greater than 3 standard deviations were noted. The outliers were then cross- checked with the original sample tags to verify whether the discrepancy was related to a clerical error during collection or preparation (i.e. duplicate labeled as a standard).

Where blanks or duplicates could not explain outliers, it was assumed that the standards test result for that particular sample had failed. Most notably are samples 102213 and N009201 which were assumed to be failed samples. The failed standards were not included in the data to generate a new mean and standard deviation; however, they were included in the following figures. Table 5-6 summarises the standards test results and control charts are presented in Desharnais et al, 2016. It can be noted that the mean for each is maintained within 2 standard deviations with only a few minor exceptions. The failures range between 0% and 2%.

It is important to note that all means and standard deviations for all standards have not been certified. Ce has also been removed from the TREO because its relative concentration far outweighs its value contribution to this deposit. The standards used in the 2011 campaign to control the data quality for that campaign had a similar performance in verification as noted by Gilbertson (2013).

Table 5-6 Summary of the Statistical Analysis for REE at the Project (2011 & 2013)



Element	Count	Observed				Warning Range		Failure Range	
		Mean	Std Dev	Min	Max	Count	Rate	Count	Rate
Ce	326	515.1	24.9	90.2	597	14	4.3%	4	1.2%
Dy	326	5.5	0.4	4.55	8.24	14	4.3%	2	0.6%
Er	326	4.2	0.3	3.39	5.9	22	6.7%	1	0.3%
Eu	326	0.5	0.1	0.15	1.94	18	5.5%	4	1.2%
Gd	326	4.0	0.4	1.65	12.5	20	6.1%	3	0.9%
Ho	326	1.2	0.1	1	1.6	10	3.1%	3	0.9%
La	326	92.5	6.6	9.3	154	8	2.5%	3	0.9%
Lu	326	0.8	0.1	0.52	0.93	19	5.8%	3	0.9%
Nb	326	381.7	29.2	34	646	9	2.8%	8	2.5%
Nd	326	26.5	3.5	4.9	85.9	23	7.1%	6	1.8%
Pr	326	11.7	0.9	1.56	26.7	11	3.4%	4	1.2%
Sm	326	4.3	0.5	1	13.4	8	2.5%	4	1.2%
Ta	326	24.6	2.5	1	37.4	14	4.3%	8	2.5%
Tb	326	0.8	0.1	0.16	1.58	10	3.1%	4	1.2%
Th	326	74.1	5.6	16.5	137	6	1.8%	3	0.9%
Tm	326	0.7	0.1	0.41	0.86	15	4.6%	3	0.9%
U	326	25.0	2.3	3.1	33.5	11	3.4%	4	1.2%
Y	326	32.6	2.8	21.3	62.4	8	2.5%	5	1.5%
Yb	326	4.9	0.3	3.1	5.8	18	5.5%	4	1.2%
TREO no Ce	326	225.3	11.5	120.7	426.3	11	3.4%	5	1.5%

5.4.4.2 Blanks

Blank material is inserted into the sample stream in order to assess any sample contamination. Tantalus inserted blank mudstone material collected from a quarry in mainland Madagascar that is known to be devoid of REE mineralisation. Blank control charts are presented in Desharnais et al, 2016.

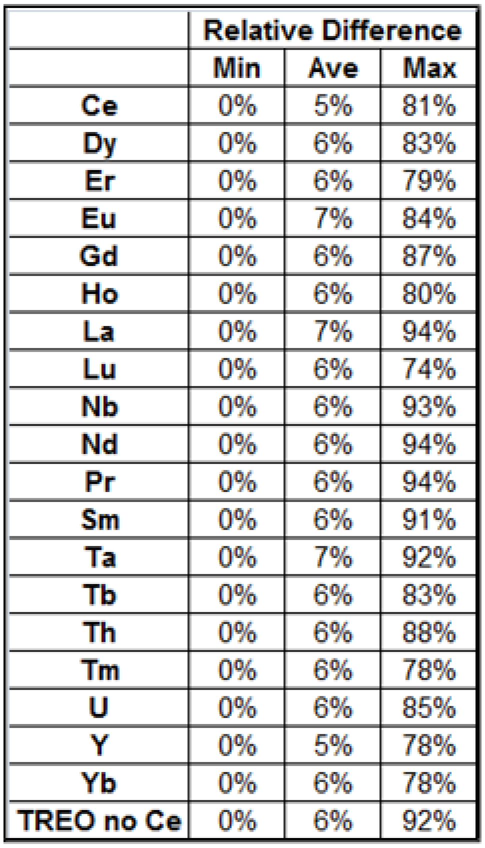
Most blanks showed consistent results with no significant bias over time. However, there were some blanks that returned higher than 3 times the detection limit, most notably N008704 and 45703 were high across all elements, and much higher than their corresponding standards. A sample switch was not identifiable from these examples, however the amplitude of the failure suggests that this is what they represent.

5.4.4.3 Duplicates

Duplicate samples are used to provide a measure of the entire error of sampling. They are collected, prepared and assayed in the same method as the originals. Pulp duplicate samples (additional half or quarter core material taken from the original core) were also sent to ALS Chemex in Vancouver to test for analytical precision at the laboratory.

569 duplicate samples (1138 total) were collected. The results of the duplicate analyses versus the original analyses are shown in Desharnais et al, 2016. The results show a good level of precision; however there are a few readings outside of the confidence level. A summary of performances for duplicates is shown in Table 5-7.

Table 5-7 Summary of Performance for Duplicates (2011 & 2013)



	Relative Difference		
	Min	Ave	Max
Ce	0%	5%	81%
Dy	0%	6%	83%
Er	0%	6%	79%
Eu	0%	7%	84%
Gd	0%	6%	87%
Ho	0%	6%	80%
La	0%	7%	94%
Lu	0%	6%	74%
Nb	0%	6%	93%
Nd	0%	6%	94%
Pr	0%	6%	94%
Sm	0%	6%	91%
Ta	0%	7%	92%
Tb	0%	6%	83%
Th	0%	6%	88%
Tm	0%	6%	78%
U	0%	6%	85%
Y	0%	5%	78%
Yb	0%	6%	78%
TREO no Ce	0%	6%	92%

5.4.5 Topographical Data

The most reliable topographic survey shows poor correlation to the handheld GPS-surveyed drillhole collars. Due to the discrepancies between the collars and the topographic surface, the collars were pressed to the best available product in order to maintain a constant baseline for which to model the geological data. The most reliable topographic map is sourced from the government and has 10m contours. SGS combined this data with the Fugro airborne survey. Together they have full coverage of the project area. See the resource estimation section for more details. SGS considers that a higher precision survey will be necessary for any economic study. A Lidar survey over the property is highly recommended.

5.4.6 Data Verification

The database was continually validated by the company on receipt of assays from the lab. Drilling, pitting, trenching and window sampling collar locations, surveys and logging was entered manually into the database by the Geologist responsible for the specific hole/pit. The data were then validated by a dedicated Database Manager.

The pit and drillhole databases were supplied to SGS as Microsoft Excel files. Prior to importation into Genesis software, the data was inspected. The data was manually validated by looking through the collar, survey, assay and lithology files for obvious errors, such as missing data or negative values.

The data importation process in Genesis incorporates its own data verification, which checks for errors in the collar, survey, assay and lithology files. The software checks for overlaps, missing data, errors in end-of-hole (EOH) depth and suspect downhole surveys.

The trench data and auger hole data were removed due to inconsistencies, together with some test pits and drillholes that were missing coordinate data. All grab sample data was removed from the database prior to modelling.

The final DB contained 4771 collars (4412 test pits and 359 drill holes), 37,212 assays and 37,212 lithologies.

5.4.7 Data Quality Summary

The sample QAQC procedures in place are, on the whole, considered appropriate for the project at its current level of development. The creation of two new standards with proper round robin certification will ensure that the accuracy of results is tested in the future. A clear protocol should be implemented by the Company to flag anomalous results from the control samples with specific steps to investigate and take actions (reanalyse batches) or accept certain results under certain conditions. Topographic data should be improved with a Lidar survey or equivalent.

It is the Author’s opinion, based on a review of all possible information, that the sample preparation and analyses used on the Project meet acceptable industry standards and the data is of sufficient quality to be used for geological and resource modeling, and estimation of Measured, Indicated and Inferred mineral resources.

5.5 Mineral Processing and Metallurgical Testing

5.5.1 Historical Testwork

The Soviet Geological Mission completed between 1988 and 1991 included the collection of samples for mineralogical and metallurgical testwork.

5.5.1.1 Soviet Mineralogical Testwork

Mineralogical testwork completed as part of the Soviet Geological Mission confirmed that the locally and historically termed fasibitikite has a granitic composition containing 30 to 50 % quartz, 10 to 30 % feldspar, 15 to 30 % riebeckite and aegirite, and up to 10 % metalliferous minerals. The identified metal-bearing minerals include pyrochlore, zircon, chevkinite, eudialyte, monazite, galena, sphalerite and magnetite. Due to the limitations of the testwork, they were unable to define the complete list of minerals that contain thorium, yttrium or tin.

The only mineral that was subject to comprehensive study was pyrochlore. Pyrochlore is found in the peralkaline granitic intrusive rocks and appears as irregularly dispersed disseminations or crystalline aggregates (0.03 to 1.5 mm). Although dispersed irregularly, pyrochlore occurs throughout the rock mass and can be concentrated at the margins of the intrusives as octahedral crystals (particularly the aegirite varieties). Weathered pyrochlore was observed to often be replaced by columbite and the typical Nb/Ta ratio for the studied samples was 13.6.

The distribution of zircon was found to be extremely irregular and to have a variable content of between 1 and 15 %. Grain size was also observed to be variable (a few hundredths of a mm to 2 mm) but with primary zircon being typically coarser and mainly found in the peralkaline granitic intrusive rocks. Secondary zircon occurs as a replacement mineral and was identified in fenite.

Chevkinite mainly occurs within the peralkaline granitic rocks whilst monazite is present in all mineralised rocks. Galena is less common and has an extremely irregular grade distribution from 100 to 6,400 ppm. Subordinate minerals were identified as xenotime, samarskite, gagarinite, sphalerite, pyrite and chalcopyrite.

The main economic elements of interest were identified as tantalum, niobium and REE (± zirconium and hafnium). Minor thorium was also identified, but in uneconomic quantities and associated with only low radioactivity.

The main Ta-Nb mineral is pyrochlore, which is often partly columbitised (where weathered) and as a result becomes more enriched in Nb. A monomineralic pyrochlore sample was calculated to contain 31.43 % Nb_2O_5, 2.31 % Ta_2O_5, 1.10 % ZrO_2, 0.35 % ThO_2 and 23.19 % TREO.

The rare earth elements were identified in chevkinite, eudialyte and pyrochlore. Cerium-bearing REEs were mainly observed in association with chevkinite, and yttrium-bearing REEs with eudialyte. The samples were determined to be LREE dominant, with particular enrichment in cerium and a notable depletion in europium - a trend that is well-documented in published literature.

5.5.1.2 Soviet Metallurgical Testwork

Metallurgical testwork completed as part of the Soviet Geological Mission included both bedrock and regolith material. The main objective of the testwork was to establish a processing methodology that would result in a rare-metal concentrate. Testwork was completed on 14 composited samples (9 bedrock and 5 regolith samples) at the OMNIC laboratory and included: Gravity concentration; Magnetic separation; and Flotation.

Flotation proved to be the most effective concentration method, with the -0.08 mm fraction containing 80 % of the minerals of interest and the -0.04 mm fraction containing 40 %. The discarded / residue material was also found to contain very fine-grained mineralisation not amenable to recovery using the utilised flotation method. Due to the limitations of the OMNIS laboratory, it was not possible to carry out further testwork on selective grinding and flotation of the fines.

5.5.2 **Pre-2013 Testwork**

Contemporary mineralogical testwork and studies have been completed in Germany by independent Geochemist Dr. Udo Jakobs (www.dr-jakobs-gmbh.de) and Consulting Geologist Dr. Thomas Hatzl (www.mineral-consult.de) and as part of research by Guillaume Estrade at the University of Toulouse in France. Contemporary metallurgical testwork has been completed in Germany by Dr. Hatzl and in Canada by the Metallurgical testwork Department of the Chemical Engineering and Applied Geochemistry section of the University of Toronto.

5.5.2.1 Mineralogical Testwork

Given the re-focus from bedrock-hosted REE mineralisation to regolith-hosted ionic adsorption-type REE mineralisation, this section describes the testwork completed on predominantly regolith material. The findings of the contemporary mineralogical studies completed on bedrock material are summarised in Section 5 - Geological Setting and Mineralisation.

In 2010, Dr. Hatzl studied a regolith sample collected from trench TANT2 in the Befitina prospect (sample TANT2-477067). The sample comprised material collected from the ferruginous zone of the regolith profile overlying syenite bedrock (Tantalus, 2012b). The sample was subject to the following analytical methods:

- X-Ray Diffraction (XRD);
- X-Ray Fluorescence (XRF);
- Fourier Transmission Infrared Spectrometry (FTIR);
- Scanning Electron Microscopy (SEM-EDX);
- Petrographic study of thin and polished sections;
- Sieving and Atterberg centrifugation (for grain-size analysis)

A summary of the XRD results for sample TANT2-477067 are provided in Table 5-8.

Table 5-8 Summary of the XRD results for Sample TANT2-477067

Fraction	*Kaolinite-D*	*Illite*	*Quartz*	*Hematite*	*Goethite*	*Gibbsite*	*Baddeleyite*
Total sample	20	n.d.	50	3	12	15	n.d.
< 2 µm	65	1	5	5	13	10	< 1
> 40 µm	8	n.d.	70	2	8	12	n.d.

All values in wt. %., n.d. = not detected

The XRD analysis indicates that half of the total sample comprises quartz, which is the dominant mineral in the coarser (> 40 µm) fraction. Kaolinite is the second most abundant mineral, and represents the most abundant mineral in the finer (< 2 µm) fraction. Both size-fractions contain significant proportions of iron (as hematite and goethite) and aluminium (as gibbsite). Interestingly, baddeleyite (ZrO2) was sufficiently concentrated in the finer fraction to be detected by XRD.

The mineralogical work classified the sample as a quartz-rich ferruginous “laterite” with a high gibbsite content and accessory baddeleyite. Petrographic studies confirmed the presence of baddeleyite and secondary zirconium, pyrochlore, rare thorianite, REE (comprising almost exclusively cerium, probably as a hydroxide/oxide) and secondary REE phosphate minerals. Zirconium was present in the coarser fraction, whilst the REE tended to occur in the finer fraction as aggregates and coatings. Secondary cerium-enriched

REE minerals represented the latest phase of the mineralisation of interest, mostly developed as very fine-grained aggregates on and between Al-Fe-hydroxides.

Based upon the mineralogical studies, the other rare earth elements appear to be hosted by relict accessory minerals including monazite, pyrochlore, thorite, and zircon, and secondary baddeleyite.

Tantalum and niobium mainly occur in minerals belonging to the pyrochlore group, with both yttropyrochlor and plumbopyrochlor observed. Both phases appear to be relict accessory minerals. In the studied sample, zirconium occurs as both relict zircon and secondary baddeleyite.

A second composite regolith sample was also mineralogically studied by Dr. Hatzl using the aforementioned methods. The sample comprised clay-rich saprolite material collected from the Caldera prospect (composite sample I679066 - I679069).

A summary of the XRD results for sample I679066 - I679069 are provided in Table 5-9.

Table 5-9 Summary of the XRD Results for Sample I679066 - I679069 (Gilbertson, 2013)

Probe	*Smectite*	*Mica*	*Illite-Smectite*	*Kaolinite-Smectite*	*Kaolinite-D*	*Chlorite*	*Quartz*	*Albite*	*K-feldspar*	*Hematite*	*Goethite*	*Gibbsite*
Total sample	1	11	7	18	21	3	33	n.d.	n.d.	2	3	1
<0.1mm A	1	9	6	13	22	2	39	n.d.	n.d.	2	3	3
<0.1mm B	1	10	4	17	22	2	34	n.d.	n.d.	2	3	4
<0.1mm C	1	10	6	18	25	2	30	n.d.	n.d.	3	3	2
0.1-0.315 mm MAG1	1	9	6	11	24	2	38	n.d.	n.d.	3	3	3
0.1-0.315 mm NONMAG1	1	5	4	2	2	1	78	<1	1	<1	<1	3
0.1-0.315 mm MID1	1	9	6	13	21	2	39	n.d.	n.d.	2	3	4
0.1-0.315 mm B’	1	4	3	5	5	1	73	n.d.	<1	1	1	5
0.1-0.5 mm	1	9	6	9	15	2	49	<1	n.d.	2	2	4
0.5-1.0 mm	1	12	6	14	17	2	42	n.d.	n.d.	2	2	2

All values in wt. %. n.d. = not detected

The XRD for the saprolite sample returned very different results to those obtained from the ferruginous zone sample. The saprolite contains a lot more clay and a greater variety of clay minerals. From an economic perspective, the presence of smectite is very significant because it has a much higher ionic exchange capacity (has the potential to adsorb more REE ions) than monomineralic kaolinite. Hematite, goethite and gibbsite are also only present in small quantities in the saprolite sample. Despite being mineralised, no REE or other rare metal bearing minerals were identified in the saprolite sample using XRD. A plausible explanation is that the mineralisation occurs as very fine-grained relict and ionic phases that were not discernible using XRD (Tantalus, 2012b).

5.5.2.2 University of Toronto Testwork

In January 2012, the University of Toronto (UoT) in Canada initiated metallurgical testwork on samples from the Project. The testwork was more specifically completed by the Department of the Chemical Engineering and Applied Geochemistry by Dr. Georgiana Moldoveanu and Prof. Vladimiros G. Papangelakis, both of whom have recently published papers specifically on the recovery of rare earth elements adsorbed on clay minerals (Moldoveanu & Papangelakis, 2012; and 2013a) and are considered to be the amongst the leading experts in this field outside of China.

The samples provided to the University of Toronto are summarised in Table 5-10.

Table 5-10 Summary of the Samples Provided to the University of Toronto (Moldoveanu, 2013)

Tantalus SampleID	*UoT Sample ID*	*Prospect*	*Type*	*From (m)*	*To (m)*	*Interval (m)*	*Material*
I618258	MC1	Caldera	Pit sample	6.50	7.00	0.50	Saprolite
I618440	MC2	Caldera	Pit sample	5.50	6.00	0.50	Saprolite
L546213	MC3	Caldera	Pit sample	4.00	4.50	0.50	Saprolite
L546571	MC4	Befitina	Pit sample	7.00	7.50	0.50	Weathered bedrock (syenite)
L547432	MC5	Befitina	Pit sample	5.00	5.50	0.50	Saprolite

MC = Madagascar Clay

The main objectives of the UoT testwork were to measure the REE and selected base metal composition of the provided samples, and investigate the leachability of the clays within the samples by measuring the REE terminal extraction under previously defined "base-line" conditions established during preceding research.

The methodology involved two phases:

Phase 1 - Clay Elemental Analysis:

- The samples (5 g each) were digested in 80 mL aqua regia (3:1 concentrated HCl:NHO_3, vol/vol) to bring the constituent elements into solution (except for the insoluble alumino-silicate matrix). The digested residue was then filtered, washed with 5% HNO_3 and denatured alcohol (85-15 % vol/vol ethanol-methanol mixture) and dried overnight in an oven at 60° C. The filtrate was then diluted to 250 mL (with DI-H_2O);
- Inductively Coupled Plasma (ICP) analysis on the solution for:
  - all lanthanide-group REE (La through Lu, plus Y);
  - Th, U, and Sc.

Phase 2 - Leaching Tests:

Batch leaching tests were performed by adding 50 g of dry sample material to 100 mL of leaching agent (i.e. Solids/Liquids = 1/2) in 250 mL Erlenmeyer flasks plugged with rubber stoppers. The flasks were equipped with Teflon-coated stirring bars and placed on a stirring magnetic plate for 30 minutes, to ensure solid suspension. At the end of the experiment, the solids were separated by filtration, washed with distilled water of pH 5 and denatured alcohol, dried in the fume hood under ambient temperature and pressure, weighted and stored for further analysis (by aqua regia digestion and ICP).

The previously defined "base-line" conditions established during preceding research (as described in Moldoveanu & Papangelakis, 2012; and 2013a) involved the following parameters:

- Lixiviants: 0.5M $(NH_4)_2SO_4$ (i.e., 1M NH_4+ exchange ions); 1M NaCl; ~ 0.5M NaCl (simulated seawater);
- S/L = 1/2 (wt/vol), i.e. 50 g clay /1/00 mL lixiviant
- Room temperature (~22°C);
- Natural pH of the system was monitored and adjusted to ~5 (with 0.1M HCl) for NaCl-based lixiviants;
- Initial test duration: 1 h (no kinetics study due to extreme difficulties in S/L separation);
- Aqua Regia Digestion (ARD) and ICP analysis were conducted on the residue (the same procedure as the one described in Phase 1) to determine the final REE and Th, U, Sc.

The UoT sample descriptions and aqua regia digestion results are provided in Table 5-11.

Table 5-11 University of Toronto Sample Descriptions and Aqua Regia Digestion Results (Moldoveanu, 2013)

UoT Sample ID	*UoT Description*	*% Dissolved during ARD*
MC1	Light brown, very fine powder	23.6
MC2	Pinkish-orange, soft chunks (easily broken with a pestle) plus some fine black sandy magnetic material	35.3
MC3	Pinkish-orange, very fine, occasional soft chunks (easily broken with the pestle)	31.5
MC4	Pinkish-orange, higher content of coarse particles (sand-like)	25.3
MC5	Brownish-orange, fine, occasional soft chunks (easily broken with a pestle)	35.7

ARD = Aqua Regia Digest

The Total Rare Earth Oxide (TREO) and Relative Rare Earth Oxide (REO) results, in wt. % are provided in Table 5-12 and Table 5-13 respectively.

Table 5-12 Total Rare Earth Oxide (TREO) Results (as wt. %) (Moldoveanu, 2013)

REO	*MC1*	*MC2*	*MC3*	*MC4*	*MC5*
La_2O_3	0.1103	0.0627	0.2047	0.0031	0.0339
Ce_2O_3	0.0476	0.0388	0.0299	0.0629	0.0204
Dy_2O_3	0.0034	0.0063	0.0066	0.0007	0.0027
Er_2O_3	0.0021	0.0036	0.0027	0.0086	0.0014
Eu_2O_3	0.0006	0.0010	0.0011	0.0000	0.0010
Gd_2O_3	0.0088	0.0097	0.0131	0.0021	0.0048
Ho_2O_3	0.0007	0.0010	0.0010	0.0019	0.0008
Lu_2O_3	0.0003	0.0007	0.0003	0.0002	0.0003
Nd_2O_3	0.0607	0.0375	0.1159	0.0028	0.0271
Pr_2O_3	0.0181	0.0112	0.0327	0.0056	0.0077
Sm_2O_3	0.0115	0.0090	0.0202	0.0009	0.0051
Tb_2O_3	0.0013	0.0014	0.0019	0.0002	0.0007
Tm_2O_3	0.0014	0.0004	0.0002	0.0007	0.0001
Y_2O_3	0.0273	0.0489	0.0362	0.0024	0.0177
Yb_2O_3	0.0013	0.0035	0.0018	0.0017	0.0013
TREO	0.295	0.235	0.468	0.093	0.125
ThO_2	0.0064	0.0079	0.0049	0.0335	0.0066
U_3O_8	0.0128	0.0283	0.0145	0.0256	0.0244
Sc_2O_3	0.0001	0.0004	0.0005	0.0002	0.0030

Total REO (TREO) content of clays is calculated as following:
Total REE “in” = sum of all individual REE in the initial clay (i.e. total mass), as detected by ICP;

Table 5-13 Relative Rare Earth Oxide (REO) Results (as wt. %) (Moldoveanu, 2013)

REO	*MC1*	*MC2*	*MC3*	*MC4*	*MC5*
La_2O_3	37.33	26.62	43.71	3.28	27.13
Ce_2O_3	16.12	16.46	6.39	67.67	16.29
Dy_2O_3	1.16	2.68	1.40	0.79	2.12
Er_2O_3	0.72	1.51	0.59	9.21	1.16
Eu_2O_3	0.22	0.41	0.24	0.01	0.78
Gd_2O_3	2.98	4.10	2.79	2.26	3.85
Ho_2O_3	0.24	0.44	0.21	2.05	0.67
Lu_2O_3	0.08	0.29	0.06	0.23	0.24
Nd_2O_3	20.56	15.93	24.75	2.98	21.69
Pr_2O_3	6.11	4.77	6.98	6.00	6.16
Sm_2O_3	3.88	3.81	4.31	0.96	4.11
Tb_2O_3	0.43	0.59	0.40	0.20	0.56
Tm_2O_3	0.46	0.16	0.05	0.74	0.08
Y_2O_3	9.24	20.75	7.74	2.61	14.12
Yb_2O_3	0.44	1.47	0.38	1.78	1.01
TREO	100	100	100	100	100

From Table 5-12 and Table 5-13 it can be observed that:

- Sample MC3 has the highest REO content, while MC4 has the lowest;
- MC1 and MC2 are rather similar in terms of total REO content relative composition;
- MC5 has less total REO content but follows similar relative distribution as MC1 and MC2;
- MC4 has the lowest REO content and seems to consist of different minerals (when compared to the other clays), with Ce, U and Th accounting for 80% of the content.
- Major REE in all clays: La, Nd, Ce, Pr, Sm and Y;

Leaching with 0.5 M (NH4)2SO4 (1M total exchange cations):

The results of leaching with 0.5 M $(NH_4)_2SO_4$ (1M total exchange cations) are provided in Table 5-14.

Table 5-14 REE Extraction Levels (as % Extraction) both as Individual REE and Total REE, Based on Solids Analysis (0.5M (NH4)2SO4, 60 min, 22°C, S/L = 1/2, pH ~ 5.4) (Moldoveanu, 2013)

REE	*MC1*	*MC2*	*MC3*	*MC4*	*MC5*
La	73.6	71.1	81.7	3.6	67.5
Ce	17.3	34.6	36.6	29.5	22.7
Dy	90.9	84.8	85.1	0.0	70.9
Er	65.4	69.9	72.1	29.1	57.3
Eu	56.8	67.7	68.8	0.0	79.1
Gd	70.6	55.6	73.2	0.0	41.6
Ho	94.7	98.2	87.1	11.0	70.0
Lu	19.9	52.5	34.4	7.9	17.7
Nd	72.3	68.9	75.2	25.4	70.5
Pr	53.6	48.7	70.5	0.0	68.6
Sm	65.2	63.5	74.9	0.0	68.3
Tb	57.5	60.3	66.1	0.0	45.1
Tm	89.0	66.5	93.9	0.0	79.4
Y	69.4	71.7	87.2	0.0	65.5
Yb	50.8	63.1	82.5	13.7	44.0
Total REE	62.4	63.0	76.1	23.4	59.4
Th	0.0	0.0	0.0	0.0	0.0
U	0.0	0.0	0.0	0.0	0.0
Sc	0.0	0.0	0.0	0.0	0.0

Total REE “in” = sum of all individual REE in the initial clay (i.e. total mass), as detected by ICP; Total REE “extracted” = the sum of all individual REE in the residue, as detected by ICP;

$$\%E = \frac{\text{Mass REE}_{\text{leached}}}{\text{Mass REE}_{\text{in clay initially}}} \text{ x } 100$$

$$\text{Mass REE}_{\text{leached}} = \text{Mass REE}_{\text{in clay initially}} - \text{Mass REE}_{\text{in final residue}}$$

General comments relating to the leaching with 0.5 M $(NH_4)_2SO_4$ (1M total exchange cations) are as follows:

Dry clays are known to absorb water (the characteristic “swelling” phenomenon). However, the clays in the Tantalus samples appeared to absorb more water compared to previous published studies with other clays (Moldoveanu & Papanagelakis, 2012; and 2013a). Due to extreme difficulties in Solid / Liquid (S/L) separation, it was not possible to collect a representative filtrate sample at the end of the experiments and recover the whole mass of clays in order to quantify the solution loss via water absorption. However, this behavior has likely been explained by the routine drying (and hence dehydration) of the samples as part of the Tantanlus sample preparation procedure.

The mass of REE leached is referenced to the final solid residue to avoid uncertainties due to lixiviant volume changes during leaching due to absorption in clay and/or sampling.

Kinetic studies were not conducted due to the difficulty of systematic sampling and S/L separating; the leaching tests were conducted for 60 minutes Based on the cited previous studies, equilibrium is usually reached in less than 15 minutes.

Leaching with NaCl-based Lixiviants:

The conditions used for the leaching of a selection of the samples are as follows:

22°C, 60 min, S/L = 1/2, initial pH of lixiviant ~ 5, adjusted with 0.1M HCl; the pH adjustment was necessary in order to avoid potential REE loss via hydrolysis (formation of insoluble hydroxides).

Based on extraction levels achieved by leaching with 0.5M ammonium sulphate, it was decided to employ only samples MC1, MC2, MC3 and MC5 for further studies (the clays with greatest leachability), as M4 demonstrated limited/low extraction.

Leaching with 1M NaCl (i.e. 1M total exchange monovalent cations available):

The results for REE Extraction during leaching with 1M NaCl are provided in Table 5-15.

Table 5-15 % REE Extraction During Leaching with 1M NaCl (Moldoveanu, 2013)

REE	*MC1*	*MC2*	*MC3*	*MC5*
La	56.0	52.4	48.8	47.9
Ce	1.0	0.5	0.0	11.1
Dy	75.6	61.5	48.6	49.1
Er	73.4	53.0	47.6	39.8
Eu	55.4	47.6	48.0	45.1
Gd	60.3	46.0	48.1	40.3
Ho	67.0	70.0	57.7	27.2
Lu	44.3	32.1	12.2	5.1
Nd	49.9	52.2	44.4	44.5
Pr	46.9	43.9	41.3	41.1
Sm	61.3	49.9	50.6	54.1
Tb	47.9	55.7	46.4	37.1
Tm	65.1	73.6	61.1	63.8
Y	55.7	57.1	48.0	48.1
Yb	42.3	48.6	41.5	39.3
Total REE	46.0	44.1	44.0	40.2

Despite the fact that both 0.5M $(NH_4)_2SO_4$ and 1M NaCl offer identical initial concentration of available exchange cations, 1M NaCl achieves lower REE extraction levels. This behavior is consistent with the hydration energy theory that was postulated during previous published work (Moldoveanu & Papanagelakis, 2012).

Leaching with Simulated Seawater Solution (SSW):

The results for REE Extraction during leaching with Simulated Seawater Solution (SSW), with ~ 0.48M Na (i.e. ~ 10.8 g/L Na^+, 19.4 g/L Cl^-, 2.7 g/L SO^{42-}, 1.28 g/L Mg^{2+}, 0.4 g/L K^+, 0.4 g/L Ca^{2-}) are provided in Table 5-16.

Table 5-16 % REE Extraction During Leaching with Simulated Seawater (0.48M Na) (Moldoveanu, 2013)

REE	***MC1***	***MC2***	***MC3***	***MC5***
La	52.4	48.0	42.8	46.0
Ce	0.0	0.0	0.0	8.4
Dy	77.7	55.4	42.7	41.8
Er	44.0	45.5	41.6	36.7
Eu	45.4	40.1	41.1	39.3
Gd	50.3	41.2	44.1	37.1
Ho	54.6	61.9	48.2	21.8
Lu	0.0	23.4	5.0	0.0
Nd	46.3	46.1	40.1	40.8
Pr	39.6	37.0	36.6	37.7
Sm	53.1	45.1	46.2	49.9
Tb	48.6	43.0	42.6	32.8
Tm	5.6	56.6	49.3	52.2
Y	50.8	51.8	42.8	45.4
Yb	38.8	41.0	34.4	33.2
Total REE	41.6	39.5	39.1	37.2

Based upon these results, simulated seawater (0.48M Na) achieves lower REE extraction levels when compared to 1M NaCl (by ~10%).

Two-Stage Leaching Experiments:

In order to investigate a possible increase of REE extraction by multi-stage leaching, a 2-stage process was applied to sample MC3 (as the material that exhibited the highest extraction levels). The leached clays were filtered, washed with DI-H_2O adjusted to pH 5 as previously explained, and re- pulped again with fresh lixiviant under identical conditions (i.e. 22°C, 60 min, S/L = 1/2, pH ~5). The utilised lixiviants comprised 0.5M $(NH_4)_2SO_4$, 1M NaCl and simulated seawater (0.48M NaCl), respectively.

In order to accelerate the data collection process, the extraction levels are solution-based and calculated with reference to the final volume. Proper extraction values should be based on solids. Nevertheless, they are comparable with solution-based ones.

$$\%\mathrm{E}_{\mathrm{total}} = \frac{\text{Mass REE}_{\text{leached total}}}{\text{Mass REE}_{\text{in clay initially}}} \text{ x } 100$$

$$\text{Mass REE}_{\text{leached total}} = \text{Mass REE}_{\text{in final solution1}} - \text{Mass REE}_{\text{in final solution2}}$$

Table 5-17 Two-stage Leaching for MC3 (22°C, 60 min, S/L = 1/2) (Moldoveanu, 2013)

Element	*0.5M $(NH_4)_2SO_4$*		*1M NaCl*		*SSW*	
	E_1	**Etot**	**E_1**	**Etot**	**E_1**	**Etot**
La	83.6	97.5	48.8	72.0	42.8	56.9
Ce	0.0	0.0	0.0	0.0	0.0	0.0
Dy	80.9	94.7	48.6	70.2	42.7	59.4
Er	86.8	93.0	47.6	66.1	41.6	57.7
Eu	62.1	77.6	48.0	67.0	41.1	51.4
Gd	82.6	96.7	48.1	70.4	44.1	59.4
Ho	75.5	93.2	57.7	80.0	48.2	63.0
Lu	52.3	61.2	12.2	12.2	5.0	5.0
Nd	80.8	94.3	44.4	65.5	40.1	52.6
Pr	75.1	87.1	41.3	60.7	36.6	48.0
Sm	90.6	94.3	50.6	74.5	46.2	61.4
Tb	84.1	98.5	46.4	66.4	42.6	56.1
Tm	53.4	60.9	61.1	77.4	49.3	57.2
Y	77.3	90.5	48.0	69.2	42.8	60.5
Yb	73.2	85.7	41.5	57.2	34.4	46.6
Total REE	76.6	88.8	44.0	64.6	39.1	52.1

As observed in Table 5-17, the two-stage leaching procedure has the ability to significantly increase overall REE extraction by an additional 10 to 20 units % (depending on the individual REE and lixiviant used).

General conclusions relating to the testwork completed by the UoT are summarised as follows:

The samples provided by Tantalus and identified as MC1 through MC5, respectively, have a content of REO ranging from 0.09 to 0.47 %wt. (as per Table 13-5);

Samples MC1, MC2, MC3 and MC5 exhibit good “ion adsorption”-type behavior (i.e. the major part of the REE content can be easily and rapidly recovered by simple leaching with either ammonium sulphate or sodium chloride solutions under ambient conditions) MC3 shows the highest leachability (76% Total REE leached), followed by MC1, MC 2 and MC5, respectively. MC4 has the lowest REE content and poor leachability (i.e. ~ 24% out of 0.09% wt. initial TREO), attributed to it comprising weathered bedrock (syenite) rather than clay-dominant material;

0.5M $(NH_4)_2SO_4$ offers the best extraction levels (Table 5-14), between 60 and 76 %, whereas 1M NaCl and simulated seawater (0.48M Na) achieve ~20 % units lower extraction levels (Table 5-15 and Table 5-16, respectively);

Individual REE extraction varies depending on sample type; The samples exhibit no extraction for U, Th and Sc;
A two-stage leaching process (i.e. leaching of previously leached clays with fresh lixiviant) on sample MC3 appears to improve the overall REE extraction levels by 10 to 20 units %, depending on the lixiviant used (Table 5-17).

Following on from the testwork conducted at UoT, the University has recommended the following course of future testwork:

- Multiple stage leaching tests using differing strengths of NaCl, $(NH_4)_2SO_4$ and a mixture of seawater and $(NH_4)_2SO_4$;
- Sedimentation and filterability testwork on the leached slurries;

- Oxalate precipitation tests, including reagent optimisation, kinetic and temperature effects, and sedimentation and filterability testwork; and
- Oxalate calcination testwork, including kinetics and final product purity.

5.5.3 Outotec Testwork

Outotec began testing in May of 2014 on samples from The Project which were to be completed in January of 2015. Results summarised here are from a preliminary report provided by Tantalus in September 2014.

Leaching experiments:
Five leaching experiments were carried out, material from one clay deposit (single sample) and material from several deposits (composite sample) were used.

Conditions and set-up:

- Solid content in the leaching = 0.5 kg clay (moist) / 1 l solution - Clay moisture content approximately 20 weight-%
- Leaching temperature = 25 – 30 °C
- Leaching time = 2 - 3 h
- Electrolyte solutions for ion-exchange leaching
    - $(NH_4)_2SO_4$, 0.50 mol/ and 0.25 mol/l
    - NaCl, 0.48 mol/l (corresponds concentrations in sea water)
    - Na_2SO_4, 0.5 mol/l
- Experimental set-up
    - Mechanically agitated tank reactor with baffles
    - Reactor size = 5 L
    - Slurry content = approximately 4 L
    - Temperature control
    - pH measurement
    - Sampling from the reactor at certain intervals to produce solution and solid samples for analysis.

Leaching was done in 0.5 mol/l $(NH_4)_2SO_4$ solution, 0.25 mol/l $(NH_4)_2SO_4$ solution, 0.48 mol/l NaCl solution (synthetic seawater). For all tests the Feed material was a single sample from one deposit and they were assayed for main metal analyses along with U, Th, Sc.

Leaching tests were also completed using a 0.5 mol/l $(NH_4)_2SO_4$ solution, on a composite sample. It also showed a rapid leaching process and low concentrations of main metals in the leach solution.

The effect of electrolyte solution on leaching of main metals showed little to no difference.

Precipitation

- Preliminary precipitation test was carried out by using oxalic acid
- Precipitate was obtained
- Na_2CO_3 was also tested as a precipitation chemical but it resulted in geltype of precipitate (clear difference compared to the precipitate obtained with oxalic acid). Oxalic acid resulted in the formation of more crystalline precipitate (however, very fine particle size also with oxalic acid).
- No precipitation of main metals in the solution. Concentrations of the main metals remained on the same level in starting solution and final solution
- REE analyses is being completed.

Process concept development and modelling
Process concept by using $(NH_4)_2SO_4$ leaching and oxalic acid precipitation as a starting case.
Main process steps:

- Leaching
- Solid-liquid separation

- Precipitation of REE
- Solid-liquid separation of REE cake
- Oxalate and gypsum removal
- Solution recycling
- Bleed stream out
- Solid residue out
- REE product (RE oxide)

Summary

- Based on the first leaching tests, the following features were observed
    - Leaching tests show very rapid leaching process (based on the main metals and REE)
    - The yields of individual REE varied a lot
    - The highest REE yields were for La (82 %) and Pr (83 %) when 0.25 mol/l $(NH_4)_2SO_4$ solution was used
    - The single sample contained low amount of REE which may have affected the REE yields. The later experiments have been carried out with composite sample which presumably contains more REE (samples in analysis).
    - Low concentrations of main metals in the leach solution (which is good feature: solution is then purer for REE precipitation, it is beneficial also for OPEX and CAPEX)
    - The original solid concentration in leaching (0.5 kg clay/ 1 l solution) seems to work in stirred tank. There is still potential to increase solid concentration in leaching step which means smaller reactor size at plant scale.
- Based on the first and preliminary precipitation test, the following features were observed
    - Use of oxalic acid seems to produce more crystalline precipitate in the REE precipitation step than Na_2CO_3
    - The main metals seem to remain in the solution during the precipitation (which is important feature: possibilities for purer product)

5.5.4 SGS Lakefield Testwork

Sample Receipt

A total of seven shipments, weighing approximately 982 kg, were received at the SGS Lakefield site between the months of April and May of 2013. These shipments consisted of 50 clay samples and 10 breccia samples which originated from the Ampasindava deposit in Madagascar. Each sample was identified by the pit number as well as the depth they were extracted from. The sample inventory is presented in Table 5-18.

Each sample was renamed Sample 1 through Sample 60 at the SGS facility.

Table 5-18 2013 SGS Lakefield Sample Inventory

SGS Sample ID	Tantalus Sample ID	Sample Type	Pit ID	Depth from	Depth to	No. of bags	Weight kg
Sample 1	TPIT 074-03100315	Breccia	TPIT074	3.1	0.0	6	34.1
Sample 2	TPIT 106-03000350	Breccia	TPIT106	3.0	3.5	6	39.0
Sample 3	TPIT 237-08001000	Breccia	TPIT237	8.0	10.0	6	36.8
Sample 4	TPIT 146-04000050	Breccia	TPIT146	4.0	5.0	6	50.6
Sample 5	TPIT 471-04000430	Breccia	TPIT471	4.0	4.1	6	32.1
Sample 6	TPIT 370-07000710	Breccia	TPIT370	7.0	7.1	6	33.0
Sample 7	TPIT 141-08000850	Breccia	TPIT141	8.0	8.5	6	36.1
Sample 8	TPIT 110-09001000	Clay	TPIT110	9.0	10.0	2	13.0
Sample 9	TPIT 114-02000300	Clay	TPIT114	2.0	3.0	2	12.2
Sample 10	TPIT 211-08001000	Clay	TPIT211	8.0	10.0	2	13.0
Sample 11	TPIT 312-09001000	Clay	TPIT312	9.0	10.0	2	14.0
Sample 12	TPIT 360-06000800	Clay	TPIT360	6.0	8.0	2	10.8
Sample 13	TPIT 011-02500450	Clay	TPIT011	2.5	4.5	2	12.8
Sample 14	TPIT 023-05000650	Clay	TPIT023	5.0	6.5	2	15.2
Sample 15	TPIT 026-05000560	Clay	TPIT026	5.0	5.6	2	13.1
Sample 16	TPIT 037-07501000	Clay	TPIT037	8.5	10.0	2	9.4
Sample 17	TPIT 088-07500800	Clay	TPIT088	7.5	8.0	2	8.9
Sample 18	TPIT 240-01000200	Clay	TPIT240	1.0	2.0	2	13.9
Sample 19	TPIT 247-07000080	Clay	TPIT247	7.0	8.0	2	14.3
Sample 20	TPIT 248-04000450	Clay	TPIT248	4.0	4.5	2	13.4
Sample 21	TPIT 249-08000900	Clay	TPIT249	7.0	9.0	2	11.5
Sample 22	TPIT 252-03000040	Clay	TPIT252	3.0	4.0	2	11.7
Sample 23	TPIT 284-06000800	Clay	TPIT284	6.0	8.0	2	14.4
Sample 24	TPIT 294-04000700	Clay	TPIT294	4.0	7.0	2	14.0
Sample 25	TPIT 303-06000700	Clay	TPIT303	6.0	7.0	2	13.0
Sample 26	TPIT 324-01000020	Clay	TPIT324	1.0	2.0	2	14.7
Sample 27	TPIT 326-04000060	Clay	TPIT326	4.0	6.0	2	13.5
Sample 28	TPIT 182-01000300	Clay	TPIT182	1.0	3.0	2	12.0
Sample 29	TPIT 192-01000200	Clay	TPIT192	1.0	2.0	2	10.2
Sample 30	TPIT 194-02000400	Clay	TPIT194	2.0	4.0	2	10.8
Sample 31	TPIT 187-02000300	Clay	TPIT187	2.0	3.0	2	10.5
Sample 32	TPIT 188-0500700	Clay	TPIT188	5.0	7.0	2	10.0
Sample 33	TPIT 060-02500350	Clay	TPIT060	2.5	3.5	2	10.5
Sample 34	TPIT 064-05000550	Clay	TPIT064	5.0	5.5	2	10.9
Sample 35	TPIT 190-07000800	Clay	TPIT190	7.0	8.0	2	13.3
Sample 36	TPIT 391-07000900	Clay	TPIT391	7.0	9.0	1	6.2
Sample 37	TPIT 421-08000900	Clay	TPIT421	9.0	10.0	3	15.9
Sample 38	TPIT 501-08000900	Clay	TPIT501	8.0	9.0	2	13.9
Sample 39	TPIT 215-05000600	Clay	TPIT215	5.0	6.0	2	10.5
Sample 40	TPIT 170-04000500	Clay	TPIT170	4.0	5.0	2	11.5
Sample 41	TPIT 488-08000900	Clay	TPIT488	8.0	9.0	2	10.7
Sample 42	TPIT 500-08001000	Clay	TPIT500	8.0	10.0	2	13.6
Sample 43	TPIT 217-02000300	Clay	TPIT217	2.0	3.0	2	13.7
Sample 44	TPIT 218-09001000	Clay	TPIT218	9.0	10.0	2	11.2
Sample 45	TPIT 259-03000400	Clay	TPIT259	3.0	5.0	2	10.9
Sample 46	TPIT 228-02000350	Clay	TPIT228	2.0	3.5	2	10.8
Sample 47	TPIT 261-04000700	Clay	TPIT261	4.0	7.0	2	10.9
Sample 48	TPIT 231-08000830	Breccia	TPIT231	8.0	8.1	6	30.9
Sample 49	TPIT 124-08000900	Clay	TPIT124	8.0	9.0	2	15.5
Sample 50	TPIT 112-03500500	Clay	TPIT112	3.5	5.0	2	12.0
Sample 51	TPIT 117-09501000	Clay	TPIT117	9.5	10.0	2	15.5
Sample 52	TPIT 104-07000800	Clay	TPIT104	7.0	8.0	2	11.2
Sample 53	TPIT 255-04000500	Breccia	TPIT255	4.0	5.0	6	32.9
Sample 54	TPIT 166-03000400	Clay	TPIT166	3.0	4.0	2	14.9
Sample 55	TPIT 280-02000500	Clay	TPIT280	2.0	5.0	2	16.5
Sample 56	TPIT 142-07000750	Breccia	TPIT142	7.0	7.5	6	33.7
Sample 57	TPIT 276-09001000	Clay	TPIT276	9.0	10.0	2	15.1
Sample 58	TPIT 270-03000600	Clay	TPIT270	3.0	6.0	2	11.5
Sample 59	TPIT 268-02000300	Clay	TPIT268	2.0	3.0	2	14.1
Sample 60	TPIT 273-04000500	Clay	TPIT273	4.0	5.0	2	11.9

SGS Lakefield Sample Preparation and Head Characterisation

Inside each pail, samples were separated in bags, identified by location and depth. Each sample was dumped on a clean floor, blended by means of cone and quartering and homogenised separately, though there were some core-like rocks in the samples, which made it difficult to take a representative sample. A 1 kg charge was split out from each sample and used for moisture determination, head assays and size-by-size characterisation.

A second 1-kg charge was taken this time from clay samples only in order to produce a master clay composite for the hydrometallurgical testing. It should be noted that at the time the master composite was blended Sample 43 through Sample 60 had not yet been received at the SGS Lakefield facility and are not part of the master clay composite. Table 5-19 presents the head assays for the Master Clay Composite plus and averaged of the head assays from the clay samples used to make the composite.

The Master Clay Composite contained a total of 2037 g/t REE, of which 17.2% are HREE. The more abundant elements are lanthanum (704 g/t), neodymium (443 g/t), cerium (342 g/t), yttrium (200 g/t) and praseodymium (129 g/t). The main gangue materials are silica (22.2%), aluminium (15.1%) and iron (6.20%).

Table 5-19 Master Clay Composite - Head Assays

Sample ID	unit	Average Sample 8-42	Master Clay Comp
H2O	%	29.5	29.8
Si	%	21.9	22.2
Al	%	15.1	15.1
Fe	%	6.11	6.20
Mg	%	0.18	0.21
Ca	%	0.03	0.03
Na	%	0.31	0.27
K	%	1.78	1.75
Ti	%	0.38	0.41
P	%	0.03	0.03
Mn	%	0.15	0.15
Cr	%	0.01	0.01
V	%	0.01	0.01
La	g/t	773	704
Ce	g/t	361	342
Pr	g/t	139	129
Nd	g/t	443	443
Sm	g/t	74.8	69.4
Eu	g/t	6.49	5.70
Gd	g/t	51.0	52.0
Tb	g/t	7.09	7.00
Dy	g/t	38.6	37.8
Ho	g/t	6.96	6.70
Er	g/t	18.7	18.9
Tm	g/t	2.42	2.50
Yb	g/t	15.6	16.3
Lu	g/t	2.39	2.40
Y	g/t	195	200
Sc	g/t	< 25	< 25
U	g/t	9.88	9.70
Th	g/t	48.0	47.6
TREE	g/t	2135	2037
LREE	g/t	1791	1687
HREE	g/t	344	349
HREE/TREE	% rel	16.6	17.2

5.5.4.1 SGS Lakefield Hydrometallurgical Testing

SGS Lakefield Shaking Tests

Since REE extraction from clay is not a leach process but an ion exchange desorption process, testing methodologies needed to be adjusted to reflect this. Each test was carried out in an Erlenmeyer flask using a wrist-shaker mechanism traditionally used for solvent extraction testwork. Standard test conditions included:

- Erlenmeyer flask in a wrist-shaker at a gentle agitation;
- Feed consisting of Master Clay Composite;
- Room temperature;
- Shaking time of 60 minutes;
- Slurry was filtered after each 60 minute contact.

A total of 28 shaking tests were conducted on the Master Clay Composite sample. They were divided into 3 separate series:

- Contact methodology (12 tests): The objective of these tests was to develop an economic method that was able to evaluate REE extraction of a large series of samples;
- Optimum eluant (12 tests): The objective of these tests was to find out the optimum eluant/conditions for REE extractions;
- Isotherms (4 tests): The objective of these tests was to develop the relationship between leach liquor (eluate) concentration and leach residue grades, much like an extraction isotherm is generated in solvent extraction (SX) testing.

SGS Lakefield Contact Methodology Tests

A total of 12 tests were carried out in order to develop a fast/practical method to evaluate REE extraction from TRE clay samples. The main test conditions for the Contact Methodology series are summarised in Table 5-20. Six different strategies were tested using two different eluants; ammonium sulphate or sodium chloride, both at 1M concentration and adjusted pH to 3.0. The main differences between the tests consisted of the number of shake contacts (from 1 to 4) and the number of displacement washes (onto the filter) using fresh eluant as wash solution. At the end of each test the wet cake was washed using 20 mL of deionised (DI) water.

Tests CM1 through CM6 consisted of only 1 shake contact. Each test cake was subsequently displacement washed with the same amount of eluant as used in the original shake contact; the differences between the tests consisted of the number of displacement washes applied and the type of eluant used. Tests CM7 through CM12 consisted of 2 to 4 shake contacts, and no eluant displacement washes.

The achieved extraction values of the Contact Methodology tests are presented in Table 5-21. The residue and solution assay data are summarised in Table 5-22 and Table 5-23, respectively. The main point was to indicate whether several contacts were required or that several displacement washes using fresh eluant were sufficient to desorb the majority of the REE from the clay, the latter saving the filtration steps between contacts, and hence, resulting in a much easier and more economic method to be applied to numerous samples. Also, a preliminary comparison between eluants was made based on the results from this series.

Table 5-20 Contact Methodology - Test Conditions

Test ID	CM1	CM2	CM3	CM4	CM5	CM6	CM7	CM8	CM9	CM10	CM11	CM12
Feed Sample	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite
Feed, g wet	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2
Feed, g dry	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0
% Solids	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%
Reagent	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4	NaCl	NaCl	NaCl	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4	NaCl	NaCl	NaCl
Reagent Conc.	1M	1M	1M	1M	1M	1M	1M	1M	1M	1M	1M	1M
Reagent pH	3	3	3	3	3	3	3	3	3	3	3	3
No. Contacts	1	1	1	1	1	1	2	3	4	2	3	4
No. Washes	1	2	3	1	2	3	0	0	0	0	0	0

Table 5-21 Contact Methodology – REE Extraction (Extraction values reported as %)

Element	CM1	CM2	CM3	CM4	CM5	CM6	CM7	CM8	CM9	CM10	CM11	CM12
Si	0	0	0	0	0	0	0	0	0	0	0	0
Al	0	0	0	0	0	0	0	1	1	0	0	0
Fe	0	0	0	0	0	0	0	0	0	0	0	0
Mg	4	6	4	3	4	3	8	7	5	3	3	5
Ca	58	67	58	55	61	64	64	62	59	54	20	48
P	5	9	13	7	7	11	9	14	14	10	11	12
Mn	1	1	2	2	1	2	2	3	3	3	1	2
La	80	83	85	80	78	87	62	88	88	68	81	85
Ce	11	11	10	13	13	15	10	29	17	8	13	13
Pr	80	84	84	77	76	85	65	88	87	64	78	81
Nd	81	85	85	75	76	84	68	88	87	64	77	80
Sm	80	83	82	73	70	82	69	88	86	61	75	78
Eu	80	82	88	77	71	84	67	91	86	64	79	77
Gd	79	82	81	74	71	81	72	88	85	63	76	77
Tb	73	75	78	71	67	78	65	84	77	56	73	71
Dy	67	69	71	66	62	71	62	80	74	54	70	67
Ho	64	66	68	62	58	67	57	77	69	48	65	63
Y	65	65	68	62	59	67	62	78	70	54	69	65
Er	58	59	59	57	51	59	53	71	64	45	62	58
Tm	53	58	55	54	49	55	44	65	57	33	56	51
Yb	49	49	49	51	44	50	42	61	53	37	54	46
Lu	47	52	45	46	41	52	38	57	52	30	54	44
Sc	1	1	2	1	2	2	1	2	2	0	2	1
Th	0	1	1	0	0	1	1	2	1	0	1	1
U	3	3	4	1	1	2	3	6	5	0	1	2

Based on the extractions achieved in these tests it can be said that both ammonium sulphate and sodium chloride are capable of extracting REE form the clay ores. LREE extractions were around 70% to 88% with the exception of Ce, which had a maximum extraction of 29% (it was mostly around 15% or less). HREE extractions were anywhere between 50% and 80%. Main elements extractions were up to 88% Nd, 50-80% Dy, 60-70% Y.

REE in the residue averaged around 600 g/t, ranging from 505 g/t REE (CM8) to 1068 g/t REE (CM10). Solution tenors were between 170 mg/L REE and 600 mg/L REE, though this numbers should not be compared since the values for the first 6 tests (CM1 to CM6) represent the PLS plus the displacement washes leading to low metal tenors in these tests.

It should be noted that in all tests, extractions for thorium, uranium and gangue elements (Si, Al, Fe, Mg, Ca, P and Mn) were very low with the exception of Ca which was around 50%, though calcium content in the clay composite sample is very low to begin with. Although Al extraction was below 1% for most of the tests, Al tenors in solution were high in comparison to REE tenors.

Maximum REE extractions were accomplished in CM3 (single contact + 3 washes, $(NH_4)_2SO_4$) and CM8 (4 contacts + no wash, $(NH_4)_2SO_4$). There appears to be slightly improved performance of ammonium sulphate versus sodium chloride, though this is not definitive at this point. Since there is no big difference between the extractions in CM3 and CM8, a single contact plus 3 displacement (eluant) washes was selected as the method to evaluate REE extraction for the remaining of the program. Worth noting is the fact that filtration was slow during all tests.

Table 5-22 Contact Methodology - Residue Assays

Element	units	CM1	CM2	CM3	CM4	CM5	CM6	CM7	CM8	CM9	CM10	CM11	CM12
Si	%	21.1	21.1	20.8	21.8	20.7	22.7	23.2	22.3	21.8	22.3	22.0	22.3
Al	%	15.0	14.8	15.5	15.3	14.6	14.6	14.1	15.6	15.5	14.9	15.3	14.8
Fe	%	6.91	5.57	6.61	6.25	8.60	6.13	5.22	5.26	6.02	6.01	6.30	6.42
Mg	%	0.17	0.17	0.15	0.17	0.17	0.22	0.25	0.14	0.18	0.21	0.18	0.16
Ca	%	0.01	0.01	0.01	0.01	0.01	0.01	0.03	0.01	0.01	0.01	0.04	0.03
P	%	0.03	0.03	0.03	0.03	0.04	0.03	0.02	0.02	0.03	0.03	0.03	0.03
Mn	%	0.15	0.18	0.14	0.12	0.19	0.13	0.10	0.10	0.12	0.15	0.26	0.14
La	g/t	129	101	96	145	163	88	200	75	88	214	140	90
Ce	g/t	297	326	377	299	284	260	414	277	299	451	275	297
Pr	g/t	24	18	18	31	33	20	34	14	17	47	31	22
Nd	g/t	80	54	64	116	111	70	109	48	57	164	114	82
Sm	g/t	14	10	12	20	23	13	16	9	10	29	21	14
Eu	g/t	1	1	1	2	2	1	2	0	1	2	2	1
Gd	g/t	10	8	10	15	17	10	11	7	8	20	15	11
Tb	g/t	2	2	2	2	3	2	2	1	2	4	2	2
Dy	g/t	12	11	11	14	17	12	11	9	11	20	14	12
Ho	g/t	3	2	2	3	3	2	2	2	2	4	3	3
Y	g/t	67	58	62	78	87	64	53	48	60	89	71	65
Er	g/t	7	7	8	9	11	8	7	6	7	11	9	8
Tm	g/t	1	1	1	1	2	1	1	1	1	2	1	1
Yb	g/t	8	7	8	8	10	8	7	7	8	11	9	9
Lu	g/t	1	1	1	1	2	1	1	1	1	2	1	1
Sc	g/t	25	25	25	25	25	25	25	25	25	25	25	25
Th	g/t	47	48	44	46	42	44	41	57	46	47	43	44
U	g/t	9	10	9	9	9	10	8	10	10	12	10	10

Table 5-23 Contact Methodology - PLS Assays

Element	units	CM1	CM2	CM3	CM4	CM5	CM6	CM7	CM8	CM9	CM10	CM11	CM12
Si	mg/L	20.2	15.3	12.6	6.9	5.1	5	20.2	19.8	16.8	10.1	8.9	11.3
Al	mg/L	173	123	91.9	85.5	56	51.8	296	273	205	113	105	113
Fe	mg/L	0.8	0.6	0.4	<0.2	<0.2	<0.2	0.3	0.3	0.3	<0.2	<0.2	0.7
Mg	mg/L	20.0	18.8	7.79	13.7	11	9.5	109	49.9	35.6	30.4	24	31
Ca	mg/L	53.7	27	13	21.7	18.3	16.6	252	88	70	66.1	37.6	91.4
P	mg/L	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5
Mn	mg/L	6.25	4.4	2.89	5.37	3.96	2.88	9.29	13.3	9.46	18.6	9.09	7.24
La	mg/L	139	92	70.3	140	93.5	79.3	171	240	252	161	173	156
Ce	mg/L	10.4	7.96	5.68	11.5	6.74	6.03	23.5	50.3	22.3	14	12.9	12.7
Pr	mg/L	26	17.1	13	25.3	17	14.7	33.2	47.1	46.5	29.8	32	28.2
Nd	mg/L	92.2	57.4	46.1	86.8	55.5	47.3	118	162	158	99.9	109	96.1
Sm	mg/L	14.9	9.09	7.4	13.9	8.99	7.51	18.7	27.5	24.5	16	18.1	14.8
Eu	mg/L	1.47	0.86	0.66	1.26	0.84	0.69	1.8	1.85	2.23	1.44	1.84	1.46
Gd	mg/L	10.9	6.89	5.51	10.6	6.78	5.82	14.5	23.1	18.6	12.3	14.3	11.8
Tb	mg/L	1.45	0.9	0.77	1.32	0.89	0.75	1.83	3.13	2.37	1.67	1.97	1.58
Dy	mg/L	6.97	4.46	3.55	6.93	4.5	3.73	9.41	16	12.1	8.2	10.1	7.94
Ho	mg/L	1.24	0.77	0.64	1.14	0.77	0.65	1.53	2.72	2.02	1.36	1.73	1.33
Y	mg/L	34.0	20.0	17.2	31.3	20.5	16.9	44.3	76.1	57.7	39.1	49.1	38.5
Er	mg/L	2.86	1.86	1.43	2.88	1.86	1.53	3.83	6.68	5.05	3.46	4.38	3.46
Tm	mg/L	0.35	0.23	0.18	0.32	0.23	0.19	0.45	0.81	0.59	0.4	0.55	0.41
Yb	mg/L	2.06	1.24	0.98	2.04	1.24	0.99	2.40	4.80	3.41	2.28	3.27	2.35
Lu	mg/L	0.29	0.18	0.14	0.28	0.18	0.14	0.34	0.65	0.49	0.31	0.47	0.34
Sc	mg/L	0.07	0.07	0.07	<0.07	<0.07	<0.07	0.08	<0.07	<0.07	<0.07	<0.07	<0.07
Th	mg/L	0.04	0.11	0.06	<0.03	<0.03	<0.03	0.08	0.12	<0.03	<0.03	<0.03	<0.03
U	mg/L	0.07	0.05	0.05	<0.02	<0.02	<0.02	0.07	0.14	0.08	<0.02	<0.02	<0.02

SGS Lakefield Optimum Eluant Tests

A total of 12 tests were conducted in order to find the best eluant/conditions for the REE extraction from weathered crust elution-deposited rare earth ores. All the tests were performed following the “1 contact plus 3 displacement washes” methodology. Different combinations of pH (from pH 2 to pH 5), eluant type and eluant concentration (from 0.5M to 1.5M) were tested in this series. Tests OE1 through OE6 used sodium chloride as eluant while tests OE7 through OE12 used ammonium sulphate. The main test conditions for the Optimum Eluant series are presented in Table 5-24. Table 5-25 shows the extractions from the Optimum Eluant tests. The residue and solution assay data are summarised in Table 5-26 and

Table 5-27, respectively. As stated before, the objective of this series was to find the optimum combination of eluant, pH and concentration to achieve the highest REE extractions.

Table 5-24 Optimum Eluant - Test Conditions

Test ID	OE1	OE2	OE3	OE4	OE5	OE6	OE7	OE8	OE9	OE10	OE11	OE12
Feed Sample	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite	Clay composite
Feed, g wet	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2	71.2
Feed, g dry	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0
% Solids	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%	33%
Reagent	NaCl	NaCl	NaCl	NaCl	NaCl	NaCl	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4	(NH4)2SO4
Reagent Conc.	1M	1M	1M	1.5M	0.5M	0.5M	1M	1M	1M	1.5M	0.5M	0.5M
Reagent pH	2	4	5	3	3	4	2	4	5	3	3	4
No. Contacts	1	1	1	1	1	1	1	1	1	1	1	1
No. Washes	3	3	3	3	3	3	3	3	3	3	3	3

Table 5-25 Optimum Eluant - Extractions (Values reported in %)

Element	OE1	OE2	OE3	OE4	OE5	OE6	OE7	OE8	OE9	OE10	OE11	OE12
Si	0	0	0	0	0	0	0	0	0	0	0	0
Al	0	0	0	0	0	0	1	0	0	1	0	0
Fe	0	0	0	0	0	0	0	0	0	0	0	0
Mg	4	4	3	3	3	5	4	3	3	4	3	4
Ca	49	39	51	12	30	66	47	44	40	40	35	35
P	18	12	19	10	13	12	12	12	11	11	12	10
Mn	2	1	1	1	1	2	3	2	1	3	2	1
La	84	90	89	89	61	57	87	86	87	87	87	87
Ce	14	11	6	12	9	8	10	12	9	7	9	9
Pr	83	88	87	88	60	56	88	87	88	89	88	88
Nd	82	87	86	87	59	54	88	85	87	87	87	86
Sm	79	84	84	84	57	55	86	84	85	85	85	85
Eu	80	83	83	86	64	54	86	85	88	86	87	86
Gd	75	81	80	81	60	57	84	83	82	84	82	82
Tb	69	74	71	74	57	54	77	77	75	79	76	76
Dy	64	69	63	67	53	52	73	70	71	73	69	70
Ho	60	64	56	62	51	48	69	66	65	69	65	65
Y	58	63	55	62	49	49	65	65	63	67	62	65
Er	51	57	50	57	45	46	61	60	58	63	59	58
Tm	48	52	40	49	39	41	53	54	51	55	50	55
Yb	44	48	39	44	38	36	52	47	46	53	43	48
Lu	38	45	37	43	32	35	48	45	44	51	46	50
Sc	2	2	2	2	2	2	3	2	2	3	2	2
Th	1	0	0	0	1	0	5	0	0	6	0	0
U	2	1	2	1	2	1	5	1	1	6	2	1

Table 5-26 Optimum Eluant - Residue Assays

Element	units	OE1	OE2	OE3	OE4	OE5	OE6	OE7	OE8	OE9	OE10	OE11	OE12
Si	%	21.8	22.4	24.9	21.2	21.7	22.2	21.0	20.2	20.1	19.6	20.6	20.9
Al	%	14.7	14.7	13.7	15.0	15.2	14.9	14.3	14.7	14.7	13.9	15.4	14.9
Fe	%	5.70	5.71	4.62	5.83	6.48	5.78	5.79	6.10	6.39	5.85	6.55	6.42
Mg	%	0.13	0.15	0.13	0.19	0.17	0.18	0.22	0.19	0.17	0.20	0.19	0.21
Ca	%	0.01	0.02	0.01	0.09	0.02	0.01	0.02	0.01	0.01	0.01	0.01	0.02
P	%	0.02	0.03	0.02	0.03	0.03	0.03	0.02	0.03	0.03	0.03	0.03	0.03
Mn	%	0.09	0.14	0.10	0.11	0.12	0.13	0.14	0.14	0.14	0.13	0.14	0.14
La	g/t	67	64	56	74	241	268	79	87	76	73	81	79
Ce	g/t	173	311	570	300	287	323	385	280	349	554	422	390
Pr	g/t	13	14	12	16	48	53	14	16	13	12	14	14
Nd	g/t	46	48	45	50	157	181	47	57	48	45	51	51
Sm	g/t	9	10	9	11	28	30	8	10	10	9	10	9
Eu	g/t	1	1	1	1	2	3	1	1	1	1	1	1
Gd	g/t	9	9	8	10	20	22	8	8	8	7	9	8
Tb	g/t	2	2	2	2	3	3	2	2	2	1	2	2
Dy	g/t	10	11	11	13	17	18	9	11	10	9	12	10
Ho	g/t	2	3	3	3	3	4	2	2	2	2	2	2
Y	g/t	59	71	71	74	95	93	60	62	65	55	68	59
Er	g/t	7	8	8	8	10	10	7	7	7	6	7	7
Tm	g/t	1	1	1	1	2	1	1	1	1	1	1	1
Yb	g/t	6	8	8	9	9	10	7	8	8	6	9	7
Lu	g/t	1	1	1	1	2	1	1	1	1	1	1	1
Sc	g/t	25	25	25	25	25	25	25	25	25	25	25	25
Th	g/t	36	48	56	51	45	45	43	44	47	42	51	45
U	g/t	9	11	10	11	10	10	9	9	11	9	11	10

Table 5-27 Optimum Eluant - PLS Assays

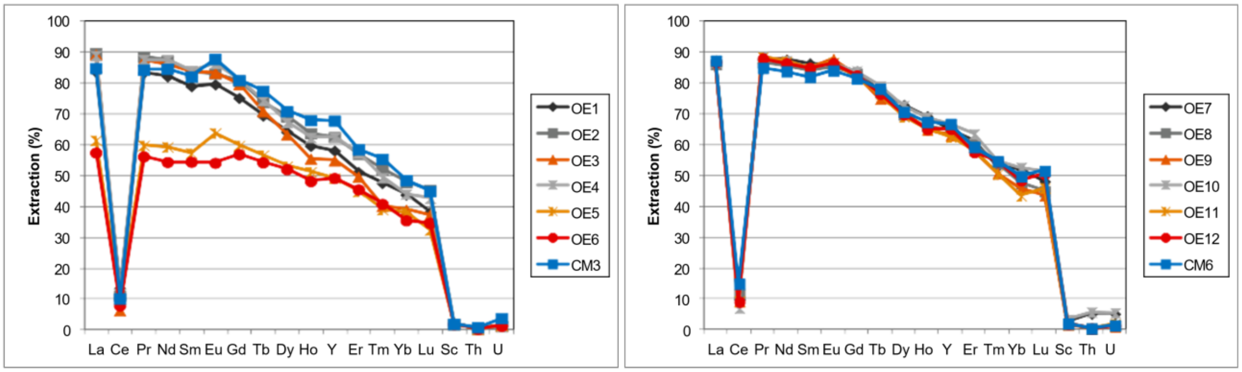
Element	units	OE1	OE2	OE3	OE4	OE5	OE6	OE7	OE8	OE9	OE10	OE11	OE12
Si	mg/L	9.3	5.6	3.8	4.6	4.2	4.8	50.2	8.4	8.4	48.2	12.1	7.4
Al	mg/L	67.7	51.4	51.4	55.7	40.8	43	236	65.1	65.1	211	98.7	62.2
Fe	mg/L	0.9	0.2	0.2	<0.2	<0.2	<0.2	36	0.2	0.2	31.1	0.3	0.2
Mg	mg/L	6.7	7.84	5.63	8.98	7.6	13	14	9.02	8.49	13.4	7.6	11
Ca	mg/L	18.3	19.3	9.2	15.4	11.4	37	30.3	16.4	15.2	15.2	10.6	16.6
P	mg/L	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5
Mn	mg/L	2.37	2.43	1.59	2.01	2.32	3.17	7.4	3.14	3.08	5.67	3.08	2.86
La	mg/L	45.1	76.3	58.1	79.1	48.0	48.6	84.7	77.6	84.0	75.6	76.3	73.1
Ce	mg/L	3.60	5.47	4.93	5.49	3.67	3.80	6.81	5.29	5.76	6.69	6.07	5.56
Pr	mg/L	8.76	14.6	10.8	14.9	8.95	9.12	16.2	15.0	16.2	15.1	14.6	14.1
Nd	mg/L	27.9	45.6	34.1	47.0	28.6	29.0	53.4	48.7	53.4	47.9	46.8	45.6
Sm	mg/L	4.60	7.40	5.46	7.60	4.67	4.79	8.57	7.84	8.63	8.1	7.72	7.38
Eu	mg/L	0.41	0.61	0.44	0.68	0.44	0.46	0.79	0.75	0.82	0.8	0.72	0.72
Gd	mg/L	3.47	5.47	3.99	5.61	3.75	3.92	6.48	5.87	6.40	5.86	5.66	5.30
Tb	mg/L	0.45	0.70	0.49	0.70	0.49	0.51	0.82	0.73	0.82	0.77	0.72	0.69
Dy	mg/L	2.32	3.54	2.43	3.49	2.46	2.64	4.04	3.64	4.04	3.8	3.56	3.41
Ho	mg/L	0.39	0.61	0.41	0.60	0.42	0.45	0.69	0.62	0.69	0.66	0.6	0.58
Y	mg/L	10.8	16.4	10.9	16.5	11.5	12.1	17.9	16.6	18.0	17.2	15.7	15.6
Er	mg/L	0.97	1.49	0.97	1.43	1.02	1.09	1.66	1.47	1.65	1.61	1.45	1.40
Tm	mg/L	0.12	0.18	0.11	0.17	0.12	0.13	0.20	0.17	0.20	0.19	0.17	0.17
Yb	mg/L	0.66	1.02	0.63	0.91	0.66	0.71	1.14	1.01	1.13	1.10	0.96	0.96
Lu	mg/L	0.09	0.15	0.09	0.13	0.09	0.10	0.15	0.13	0.15	0.15	0.13	0.13
Sc	mg/L	<0.07	<0.07	<0.07	<0.07	<0.07	<0.07	0.11	<0.07	<0.07	0.14	<0.07	<0.07
Th	mg/L	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	0.38	<0.03	<0.03	0.41	<0.03	<0.03
U	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	0.08	<0.02	<0.02	0.08	0.03	<0.02

Figure 5-14 shows the REE extractions for the Optimum Eluant series; the graph on the left shows the extractions from the tests in which sodium chloride was used as eluant while the graph on the right shows the tests where ammonium sulphate was used. Also noticed that CM3 and CM8 extractions were also plotted in Figure 5-14 for comparison reason (these tests were performed using the "1 contact plus 3 displacement washes" methodology, too).

As shown in Figure 5-14 (right) REE extractions were very similar in all the tests were ammonium sulphate was used as eluant despite the fact that different concentrations of ammonium sulphate were tested as well as different pH values. Main elements extractions were 85-88% Nd, 69-73% Dy, 62- 67% Y. The only difference between them was in terms of Th/U extractions; although mostly zero for most of the tests Th/U extractions increased up to 5-6% when the pH was adjusted to its lowest value (OE7, pH 2) and when the ammonium concentration was at its highest (OE10, 1.5M). Also, there are no visible differences when working at 1M ammonium sulphate and pH values of 3, 4 and 5 (CM6, OE8 and OE9) meaning that the process is robust and can handle ups and downs in pH with no repercussions in performance (i.e. REE extractions).

Based on these results, a concentration of 1M ammonium sulphate at an adjusted pH of 4 were selected as the best conditions when working with ammonium sulphate as eluant.

Figure 5-14 Optimum Eluant REE Extractions



Tests where sodium chloride was used as eluant (Figure 5-14, left) REE extractions were more sensitive to changes in process conditions than when ammonium sulphate was used. This indicates that the system is more susceptible to changes, specifically changes in sodium chloride concentrations. When the sodium chloride concentration was decreased to 0.5M (OE5 and OE6) there was a significant drop in REE extractions, around 30% for LREE and between 10-20% for the rest.

An increment on sodium chloride concentration from 1M (CM3) to 1.5M (OE4) did not have any remarkable improvements in REE extractions. Also, there were no visible differences when working at 1M sodium chloride and pH values of 3, 4 and 5 (CM3, OE2 and OE3); main elements extractions during these tests were 86-87% Nd, 63-69% Dy, 55-63% Y.

Based on these results, a concentration of 1M sodium chloride at an adjusted pH of 4 were selected as the best conditions when working with sodium chloride as eluant.

Since the ammonium sulphate system appeared more robust than the sodium chloride system the isotherm tests were performed using ammonium sulphate as eluant.

SGS Lakefield Isotherms Tests

A total of 4 tests were performed in order to build an isotherm for the desorption process for the REE extraction from weathered crust elution-deposited rare earth ores. These tests were carried out using 1 contact (1M ammonium sulphate at pH 4) and no eluant wash; the solids were subjected to one DI water wash after contact. Different ore-eluant ratios were tested in this series. The main test conditions for the Isotherms series are summarised in Table 5-28. The extractions plus residue and solution assay data are presented in Table 5-29.

Table 5-28 Isotherms - Test Conditions

Test ID	IS1	IS2	IS3	IS4
Feed Sample	Clay composite	Clay composite	Clay composite	Clay composite
Feed, g wet	42.7	71.2	142.5	284.9
Feed, g dry	30	50	100	200
% Solids	9%	17%	50%	67%
Reagent	$(NH_4)_2SO_4$	$(NH_4)_2SO_4$	$(NH_4)_2SO_4$	$(NH_4)_2SO_4$
Reagent Conc.	1M	1M	1M	1M
Reagent pH	4	4	4	4
No. Contacts	1	1	1	1
No. Washes	0	0	0	0

Table 5-29 Isotherms - Extractions, Residue and Solution Assays

Parameter	Extractions (%)				Residue assays (%, g/t)				Solution assays (mg/L)			
Element	IS1	IS2	IS3	IS4	IS1	IS2	IS3	IS4	IS1	IS2	IS3	IS4
Si	0	0	0	0	21.4	21.5	21.6	21.8	5.60	10.7	24.9	34.4
Al	0	0	0	0	15.5	15.4	15.1	15.1	53.5	91.5	289	478
Fe	0	0	0	0	6.31	6.28	6.65	6.28	0.20	0.20	0.20	0.20
Mg	4	4	4	4	0.18	0.17	0.18	0.17	7.10	14.3	56.0	80.3
Ca	33	53	36	37	0.02	0.01	0.01	0.03	10.5	16.8	63.1	159
P	20	10	4	3	0.02	0.03	0.03	0.03	<5	<5	<5	<5
Mn	1	1	1	2	0.15	0.14	0.15	0.14	2.12	4.36	14.8	29.7
La	85	82	77	71	95	114	157	242	54.2	111	422	769
Ce	13	11	10	10	281	317	338	344	4.24	8.03	28.7	49.4
Pr	84	82	77	71	18	22	29	44	10.0	21.0	79.9	140
Nd	85	83	78	73	59	66	92	134	33.9	70.5	267	474
Sm	81	79	74	69	12	14	18	24	5.30	11.1	41.9	72.3
Eu	83	80	78	67	1	1	1	2	0.51	1.12	4.00	6.13
Gd	80	80	76	74	10	10	12	15	4.09	8.56	32.1	55.0
Tb	75	76	70	68	2	2	2	3	0.54	1.13	4.23	7.10
Dy	70	68	65	64	11	12	14	15	2.74	5.53	20.7	34.3
Ho	63	64	62	59	3	3	3	3	0.45	0.95	3.55	5.69
Y	64	64	62	60	66	67	72	78	12.1	25.5	97.0	157
Er	57	58	56	55	8	8	8	8	1.07	2.33	8.53	13.6
Tm	51	49	47	49	1	1	1	1	0.13	0.27	1.03	1.65
Yb	49	47	46	47	8	8	8	8	0.75	1.57	5.74	9.32
Lu	45	45	40	43	1	1	2	1	0.11	0.21	0.83	1.25
Sc	3	2	1	1	25	25	25	25	0.07	0.07	0.07	0.07
Th	1	0	0	0	42	43	43	42	0.03	0.03	0.04	0.04
U	3	2	1	1	9	9	9	10	0.02	0.03	0.09	0.12

As it is shown in Figure 5-15 when the eluant:ore ratio decreased REE extractions also decreased; at the highest eluant:ore ratio (IS1, 10:1) the main elements extractions were 85% Nd, 70% Dy and 64% Y while at the lowest eluant:ore ratio (IS4, 1:2) these extractions were 73% Nd, 64% Dy and 60% Y. It should be noted that the difference in extractions is more significant for the LREE while HREE extractions are similar regardless of the eluant:ore ratio used.

Figure 5-15 Isotherms REE Extractions

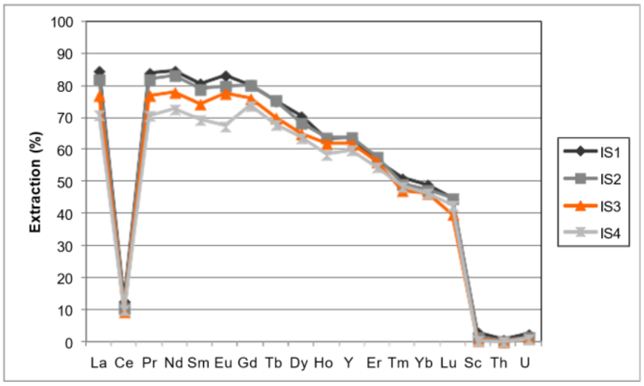
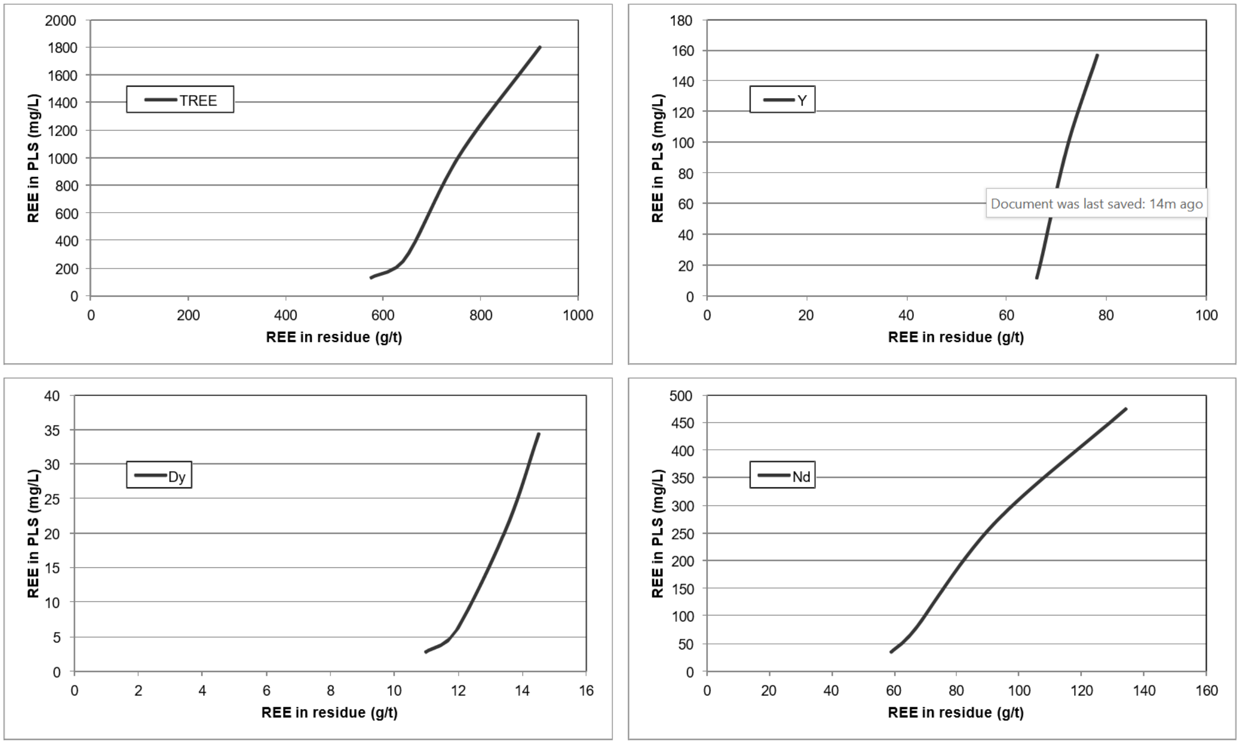


Figure 5-16 shows the isotherms for TREE and the main elements. When the eluant:ore ratio decreased REE in solution increased considerably from 130 mg/L REE (IS1, 10:1) up to 1796 mg/L REE (IS4, 1:2) even though extractions only decreased by 10%. REE in the residue also increased from 575 g/t REE up to 922 g/t REE, though this is mainly due to the LREE (La, Ce, Pr, Nd and Sm) since the rest only increased from 110 g/t HREE up to 134 g/t HREE.

These results suggested that using a counter-current circuit with a low eluant:ore ratio can achieve high REE tenors in solution while maintaining the same level of REE extraction seen in previous results.

Figure 5-16 TREE, Y, Nd and Dy Isotherms



5.5.4.2 SGS Lakefield Heap Leaching Tests

SGS Lakefield Tests Design and Feed Preparation

After all three series of shaking tests were finalised two column (heap) leach tests were designed. The main goals were to study the physical behaviour of the column (irrigation, compaction, etc.) and to confirm the results from the shaking tests. Each test was carried out in a clear-PVC column with a 508 mm internal diameter and 1.83 m in height. Based on the SGS Lakefield Optimum Eluant Tests results Column 1 was run using 1M ammonium sulphate solution adjusted to pH 4 as eluant while Column 2 used 1M sodium chloride solution adjusted to pH4. Standard test conditions included:

- Feed consisting of Master Clay Composite;
- Irrigation rate of 15 L/h/m^2 (equivalent to 0.5 mL/min);
- Room temperature;
- Running time of 218 hours;
- DI water washing at 60 L/h/m^2 for 24 hours.

Before charging to the column, the feed for each column was agglomerated using their respective eluant solution as binding agent. This was achieved by spraying eluant onto the feed and rolling the sprayed clay on a plastic sheet in doses until the feed began to form agglomerates of material that were not immediately broken by physical force. Once sufficiently wet, the feed was allowed to air dry. Photographs of each agglomerated feed are shown in Figure 5-17. Although not on purpose, Column 1 feed was dried for 1 extra day in order to start both column tests at the same time.

Figure 5-17 Column Tests Agglomerated Feed - Column 1 (left) and Column 2 (right)

SGS Lakefield Column Operation

Figure 5-18 shows a photograph of the set up used for the column tests. Once agglomerated, the feed was slowly added to the columns to avoid breaking the agglomerates; columns were tapped during this process to ensure uniform packing of the column. Each column was weighed before and after adding the feed as well as at the end of the test. Eluant addition was started immediately, considered as time zero. Discharge was not controlled, i.e. it was not pumped out of the columns.

Figure 5-18 Photograph of Column Set-up

Table 5-30 shows the main physical parameters for the Heap leaching tests. Tests details are included in Appendix C. There were significant differences in the physical behavior of the solids, starting from lower moisture for Column 1 solids at the beginning of the tests. Since the same clay composite was used in both Column 1 and 2 and the amount of solution used to agglomerate the solids was similar (around 100 mL), the difference in moisture is attributed to the drying time.

Another difference between the two systems is the time it took the column to produce the first discharge. It only took 2 hours to start collecting discharge from Column 2 while Column 1 discharge took 26 hours to start. Observations were made that the solids in Column 1 seemed to be soaking up the eluant (based on change of appearance; dry to wet) and that could be the reason why it took 24 hours more than Column 2 to start discharging.

Over the course of the test, the solid bed height within Column 1 was observed to slump 20 mm while the solids in Column 2 did not collapse. This also seems to be related to the difference in feed moisture at the beginning of the test; the less wet the solids are the less compactable they will be. Once the solids in Column 1 got soaked in they started to collapse and the final height was achievable right after the first discharge showed up.

Overall average feed and discharge rates were calculated using the mass differences on the weighed containers. The average eluant feed rate was slightly lower than target at 14.1 L/h/m^2 and 12.8 L/h/m^2 for Column 1 and Column 2, respectively. This is due to the drift inherent in any pump calibration curve, exacerbated by the very slow flowrate required. The average discharge rate (taken from the time of first discharge onwards) was calculated to be 13.6 L/h/m^2 and 12.8 L/h/m^2 for Column 1 and Column 2 respectively, slightly less than the feed rate due to entrainment of eluant within the column.
After 218 hours of running time the addition of eluant was stopped and the columns were allowed to drain. Once they stopped draining the solids were washed with DI water at an irrigation rate of 60 L/h/m^2, even though the washing irrigation rate was 4 times bigger than the eluant irrigation rate none of the columns presented any operational problem to handle such a fast rate, i.e. not compaction of the solids or flooding was observed.

Table 5-30 Heap Leaching - Main Parameters

Parameter	Unit	CL1	CL2
Feed Sample	-	Clay composite	Clay composite
Reagent	-	$(NH_4)_2SO_4$	NaCl
Reagent Conc.	mol/L	1.0	1.0
Reagent pH	-	4.0	4.0
Wet Feed	g	3080	3681
Dry Feed	g	2502	2584
Moisture	%wt	19%	30%
Initial height	cm	154	150
Initial Bulk SG	-	0.987	1.211
Run time	h	218	218
Washing time	h	24	24
First discharge*	h	26	2
Eluant added	mL	6672	5882
Discharge collected	mL	5360	5481
Avg. Feed Rate	$L/h/m^2$	14.1	12.8
Avg. Discharge Rate	$L/h/m^2$	13.6	12.8
Wet residue	g	3701	3746
Dry residue	g	2454	2575
Moisture	%wt	34%	31%
Final height	cm	134	150
Final Bulk SG	-	1.363	1.232

SGS Lakefield Chemical Performance

Throughout these tests samples of the discharge were collected, weighed and assayed. Also, samples from the washed and the final solids were analyzed for the same elements. Table 5-31 and Table 5-32 show the calculated extractions for Column 1 and Column 2, respectively. Extractions were calculated based on units of metals in solution (discharge plus wash) and the total units of metals (discharge plus wash plus residue).

Table 5-31 Column 1 - Extractions (Values reported as %)

Element	31 h	35 h	38 h	50 h	62 h	74 h	98 h	122 h	146 h	170 h	194 h	218 h
Si	0	0	0	0	0	0	0	0	0	0	0	0
Al	0	0	0	0	0	0	0	0	0	0	0	0
Fe	0	0	0	0	0	0	0	0	0	0	0	0
Mg	0	1	1	3	4	4	4	4	4	4	4	4
Ca	1	2	4	10	12	14	15	15	15	16	16	16
P	0	0	0	1	1	1	2	3	3	4	5	8
Mn	0	0	0	1	2	2	2	2	2	2	2	2
La	5	12	21	57	69	74	79	81	83	83	84	86
Ce	1	2	4	9	11	12	13	13	14	14	14	14
Pr	4	11	21	60	72	76	81	83	84	85	85	86
Nd	5	11	21	62	75	80	83	85	86	86	87	88
Sm	5	12	22	64	76	80	82	83	84	84	85	85
Eu	5	13	23	66	78	82	84	85	86	86	86	87
Gd	5	13	24	67	79	82	84	85	85	85	86	86
Tb	5	12	22	61	72	75	77	77	78	78	78	79
Dy	4	11	20	58	68	70	72	73	73	73	73	73
Ho	4	9	18	52	61	64	65	66	66	66	67	67
Y	4	9	18	53	62	64	66	66	66	66	67	67
Er	3	8	16	48	57	59	60	61	61	61	61	62
Tm	3	7	13	42	50	52	53	54	54	54	55	57
Yb	2	6	12	41	49	51	52	53	53	53	53	54
Lu	2	6	11	37	45	47	48	48	49	49	49	51
Sc	0	0	0	0	0	0	0	0	0	1	1	1
Th	0	0	0	0	0	0	0	0	0	0	0	0
U	0	0	0	1	1	2	2	2	2	2	2	3

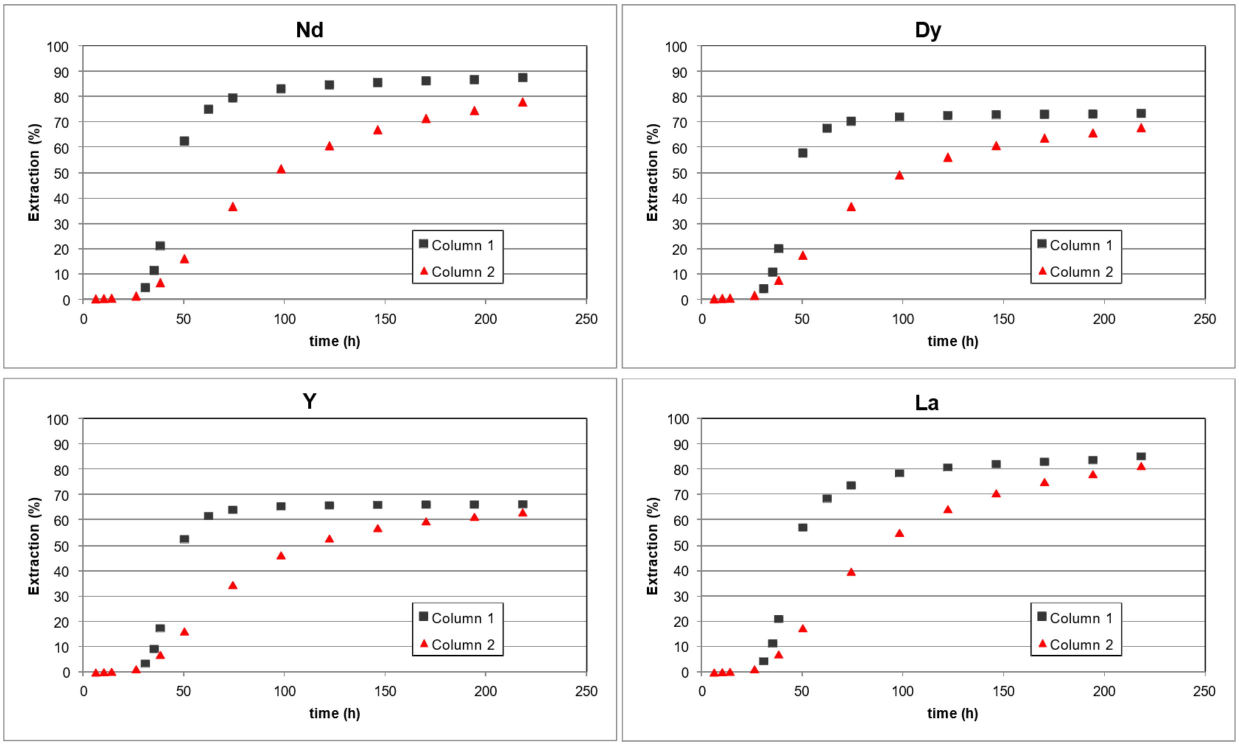
Table 5-32 Column 2 - Extractions (Values reported as %)

Element	6 h	10 h	14 h	26 h	38 h	50 h	74 h	98 h	122 h	146 h	170 h	194 h	218 h
Si	0	0	0	0	0	0	0	0	0	0	0	0	0
Al	0	0	0	0	0	0	0	0	0	0	0	0	0
Fe	0	0	0	0	0	0	0	0	0	0	0	0	0
Mg	0	0	0	1	2	2	4	4	4	4	4	4	4
Ca	0	1	1	2	6	11	16	19	20	20	20	21	21
P	0	0	0	0	1	1	2	2	3	3	4	4	7
Mn	0	0	0	0	1	1	1	1	1	1	1	1	2
La	0	0	0	1	7	18	40	55	65	71	75	78	82
Ce	0	0	0	0	1	2	6	8	9	10	11	11	12
Pr	0	0	0	1	7	16	38	52	61	68	72	75	79
Nd	0	0	0	1	7	16	37	52	61	67	71	75	78
Sm	0	0	0	1	7	16	36	50	58	64	68	71	74
Eu	0	0	0	1	8	18	39	53	62	67	71	74	77
Gd	0	0	0	2	8	19	40	54	62	68	71	74	77
Tb	0	0	0	2	8	18	38	51	59	64	67	69	72
Dy	0	0	0	1	7	17	37	49	56	61	64	66	68
Ho	0	0	0	1	7	16	34	45	52	56	59	61	63
Y	0	0	0	1	7	16	35	46	53	57	60	62	63
Er	0	0	0	1	7	15	32	43	49	53	55	57	59
Tm	0	0	0	1	6	13	28	37	43	46	48	50	53
Yb	0	0	0	1	6	13	27	36	41	45	47	48	50
Lu	0	0	0	1	5	11	25	33	38	41	43	44	46
Sc	0	0	0	0	0	0	0	0	0	0	0	1	1
Th	0	0	0	0	0	0	0	0	0	0	0	0	0
U	0	0	0	0	0	0	0	0	0	0	0	0	1

REE extractions were higher in Column 1 (ammonium sulphate) than in Column 2 (sodium chloride); main metal extractions were 88% Nd, 73% Dy, 67% Y, 86% La for Column 1, and 78% Nd, 68% Dy, 63% Y and 82% La for Column 2. Not only were the extractions lower in Column 2 they also took more time to achieve those extractions as is shown in Figure 5-19. This might be related to the fact that due to the lower moisture in Column 1 feed the eluant had to displace a smaller amount of water than in Column 2.

Gangue extractions as well as Th and U extractions remained low in Column 1 and Column 2

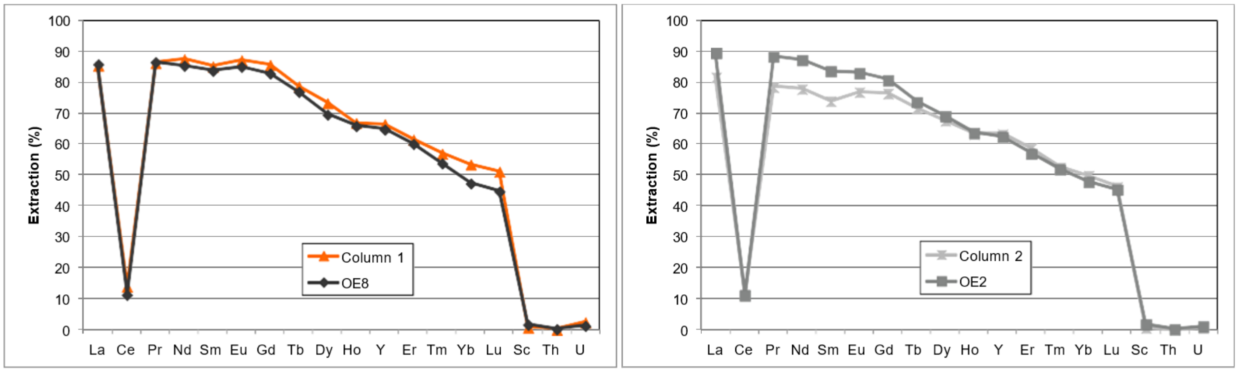
Figure 5-19 Heap Leaching - Main Elements Extractions vs. Time



As stated before, one of the main objectives of the Heap Leaching tests was to compare REE extractions obtained in the column tests against REE extractions achieved in the shaking tests. Column 1 and Column 2 REE final extractions are plotted in Figure 5-20 along with OE8 and OE2 extractions (from the SGS Lakefield Optimum Eluant Tests).

When comparing the tests using ammonium sulphate as eluant (Figure 5-20, left) it can be seen that Column 1 and OE8 REE extractions were very similar; if any Column 1 extractions were slightly higher meaning that the highest REE extractions possible were achieved during Column 1 run. Gangue material (Si, Al, Fe, Mg, Ca, P, Mn) and Th, U extractions were also as low as reported for OE8 test.

Figure 5-20 Heap Leaching Extractions vs. Optimum Eluant Extractions



When comparing the tests using sodium chloride as eluant (Figure 13-7, right) it can be seen that Column 2 REE extractions were lower than OE8 REE extractions, more distinguishable for the light and medium REE (La to Tb).

As is shown in Figure 5-21, the discharged PLS reached a maximum tenor at or around the 50 hour mark for the two columns even though Column 1 started discharging 24 hours later than Column 2. After that point the amount of REE in solution begins to fall. At the maximum, the discharge PLS was approximately 4700 mg/L TREE in Column 1 and 1500 mg/L in Column 2. It seems that after 50 hours the majority of the REEs available for desorption in Column 1 have been extracted as the REE tenors in the discharge PLS had a remarkable drop right after that. It is more difficult to make the same statement for Column 2 since the drop in REE tenors is smoother suggesting that there was extraction after the 50-hour mark. For more details the discharge PLS, wash and residue assays for Column 1 and Column 2 are shown in Table 5-33 and Table 5-34, respectively

Figure 5-21 Heap Leaching - Column 1 (left) and Column 2 (right) Discharge TREE tenors

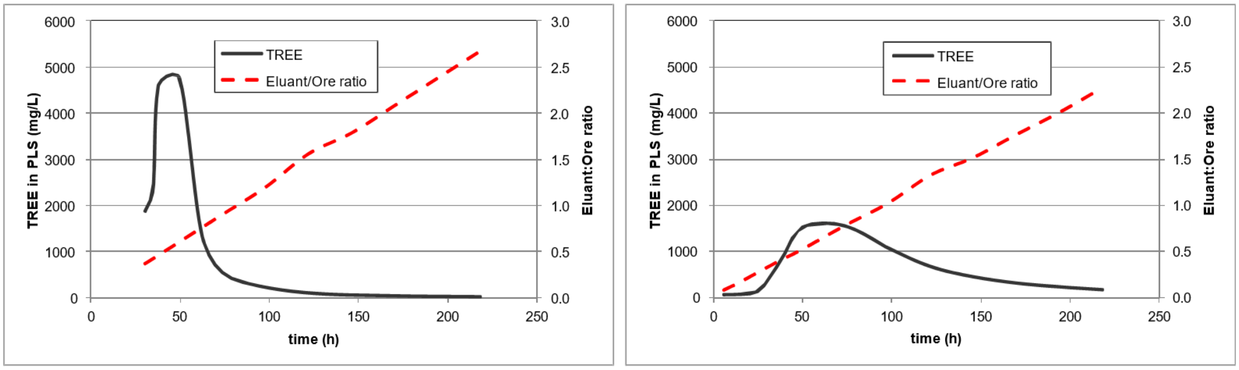


Table 5-33 Column 1 - Discharge, Wash and Residue Assays

Sample ID	31 h (mg/L)	35 h (mg/L)	38 h (mg/L)	50 h (mg/L)	62 h (mg/L)	74 h (mg/L)	98 h (mg/L)	122 h (mg/L)	146 h (mg/L)	170 h (mg/L)	194 h (mg/L)	218 h (mg/L)	Wash (mg/L)	Residue (% or g/t)
Si	44.2	44.6	47.0	43.0	30.4	23.0	17.8	14.2	12.2	11.4	10.7	10.2	6.00	23.1
Al	314	419	926	1600	573	215	91.4	44.7	29.0	23.0	20.4	17.9	3.20	14.6
Fe	0.20	0.30	0.30	0.30	0.20	0.20	0.20	0.20	0.20	0.20	0.30	0.30	0.20	5.50
Mg	108	147	246	274	67.7	14.9	3.49	0.75	0.24	0.11	0.11	0.10	0.07	0.17
Ca	152	199	339	334	120	57.0	27.0	12.4	6.90	4.70	4.20	3.40	2.00	0.06
P	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	0.02
Mn	36.7	55.9	86.0	91.9	27.7	8.70	3.29	1.47	0.97	0.80	0.74	0.70	0.16	0.15
La	792	981	1910	1830	575	257	119	58.6	33.7	22.5	16.3	12.2	6.28	99.0
Ce	65.9	82.5	149	140	44.1	19.5	9.30	4.88	3.04	2.10	1.60	1.23	0.50	279
Pr	143	180	351	363	111	44.9	19.5	9.19	5.50	3.72	2.78	2.08	0.83	17.4
Nd	474	587	1160	1280	383	136	54.3	24.6	14.3	9.51	6.99	5.22	1.91	52.0
Sm	77.3	100	197	214	58.5	18.7	6.65	2.78	1.57	0.99	0.76	0.56	0.18	10.1
Eu	7.13	8.58	16.5	17.4	4.81	1.44	0.47	0.20	0.11	0.07	0.05	0.04	0.03	0.70
Gd	63.8	81.7	158	166	42.6	12.3	3.82	1.30	0.66	0.42	0.27	0.21	0.06	7.30
Tb	8.21	10.3	20.0	20.7	5.43	1.57	0.49	0.18	0.10	0.05	0.04	0.03	0.03	1.50
Dy	39.5	51.4	101	107	27.5	7.67	2.30	0.82	0.43	0.24	0.16	0.13	0.05	10.2
Ho	6.40	8.21	16.2	17.9	4.59	1.29	0.38	0.14	0.07	0.05	0.03	0.02	0.02	2.30
Y	176	226	451	498	128	33.3	9.40	3.04	1.34	0.72	0.44	0.31	0.08	64.0
Er	14.7	19.4	39.6	45.4	11.5	3.10	0.92	0.32	0.14	0.09	0.07	0.05	0.04	7.20
Tm	1.69	2.23	4.59	5.46	1.41	0.41	0.12	0.05	0.04	0.04	0.04	0.04	0.04	1.10
Yb	9.13	11.9	25.1	31.0	8.22	2.34	0.70	0.26	0.14	0.08	0.07	0.05	0.02	6.80
Lu	1.22	1.59	3.42	4.40	1.22	0.31	0.10	0.04	0.03	0.03	0.03	0.03	0.03	1.10
Sc	0.07	0.07	0.12	0.19	0.12	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	25.0
Th	0.03	0.04	0.10	0.21	0.11	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	44.9
U	0.12	0.14	0.27	0.50	0.34	0.17	0.09	0.05	0.04	0.03	0.03	0.02	0.02	9.30

Table 5-34 Column 2 - Discharge, Wash and Residue Assays

Sample ID	6 h (mg/L)	10 h (mg/L)	14 h (mg/L)	26 h (mg/L)	38 h (mg/L)	50 h (mg/L)	74 h (mg/L)	98 h (mg/L)	122 h (mg/L)	146 h (mg/L)	170 h (mg/L)	194 h (mg/L)	218 h (mg/L)	Wash (mg/L)	Residue (% or g/t)
Si	9.00	8.40	8.50	11.5	16.8	21.0	17.9	14.8	12.3	11.2	9.70	8.60	8.20	3.10	22.7
Al	22.9	23.9	25.8	67.2	300	466	399	263	150	103	72.0	56.5	44.6	5.70	14.9
Fe	0.20	0.20	0.20	0.30	0.30	0.30	0.40	0.20	0.20	0.20	0.20	0.20	0.20	0.20	6.10
Mg	34.9	35.2	37.7	66.1	135	134	77.6	31.8	11.7	5.04	2.23	1.17	0.64	0.08	0.17
Ca	53.8	51.1	54.6	104	251	277	178	77.9	31.5	14.4	6.60	4.00	2.40	0.90	0.06
P	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	0.02
Mn	12.7	12.9	13.8	24.3	46.1	45.8	23.6	10.4	4.31	2.35	1.45	1.07	0.86	0.15	0.17
La	20.5	21.8	23.7	64.0	336	638	657	450	281	186	130	93.9	67.6	6.49	129
Ce	1.38	1.53	1.68	4.60	25.6	51.0	53.9	37.7	23.5	15.7	10.9	7.94	5.80	0.60	355
Pr	3.56	3.82	4.15	11.0	60.0	113	118	82.3	52.2	35.5	24.5	18.1	13.2	1.28	28.8
Nd	11.9	12.7	13.6	36.1	193	370	385	278	173	119	82.2	61.2	44.4	4.47	99.0
Sm	1.89	2.04	2.19	5.87	31.4	59.4	60.6	41.5	25.7	17.2	12.2	8.77	6.52	0.68	18.8
Eu	0.20	0.20	0.22	0.58	3.00	5.40	5.37	3.63	2.16	1.43	0.97	0.73	0.50	0.05	1.40
Gd	1.80	1.77	1.91	5.29	27.9	50.3	46.9	32.0	18.5	12.2	8.02	5.98	4.23	0.42	12.6
Tb	0.23	0.27	0.27	0.72	3.68	6.57	6.16	4.10	2.44	1.57	1.03	0.72	0.48	0.05	2.10
Dy	1.25	1.22	1.31	3.59	18.9	33.9	31.2	20.3	11.6	7.57	4.79	3.41	2.38	0.24	12.6
Ho	0.21	0.21	0.23	0.64	3.18	5.61	5.26	3.42	2.04	1.29	0.85	0.58	0.39	0.05	2.60
Y	5.69	5.87	6.39	17.7	90.5	159	150	96.1	55.0	34.0	22.0	15.0	10.2	0.95	72.0
Er	0.53	0.48	0.55	1.53	7.90	13.9	12.6	8.13	4.65	3.12	1.94	1.33	0.89	0.10	7.50
Tm	0.06	0.06	0.06	0.18	0.94	1.63	1.54	1.01	0.59	0.38	0.23	0.16	0.11	0.04	1.20
Yb	0.34	0.37	0.37	1.13	5.14	9.13	8.39	5.58	3.18	2.06	1.30	0.91	0.65	0.06	7.20
Lu	0.05	0.05	0.06	0.15	0.71	1.26	1.22	0.77	0.45	0.29	0.19	0.13	0.08	0.03	1.20
Sc	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	25.0
Th	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	42.1
U	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	8.50

5.5.4.3 SGS Lakefield Liquor Treatment Tests

With the solution collected from Column 1 (ammonium sulphate) during the heap leaching tests, a series of scoping level liquor treatment tests were run. The feed for these tests was a composite of different discharge PLS from Column 1 aiming to obtain a desirable REE tenor. A sample of this composite was submitted for analysis, the assay data are presented in Table 5-35.

These tests were divided in two groups: oxalic acid precipitation tests, aiming to selectively precipitate the REE, and aluminium removal tests, aiming to remove aluminium from the heap leach discharge PLS while keeping most, if not all, the REE in the remaining solution.

Table 5-35 Liquor Treatment - Feed Assays

Element	Si mg/L	Al mg/L	Fe mg/L	Mg mg/L	Ca mg/L	P mg/L	Mn mg/L	La mg/L	Ce mg/L	Pr mg/L	Nd mg/L	Sm mg/L	Eu mg/L
CL1 composite	27.7	611	<0.2	89.7	150	<5	34.8	745	57.9	143	506	74.7	6.6

Element	Gd mg/L	Tb mg/L	Dy mg/L	Ho mg/L	Y mg/L	Er mg/L	Tm mg/L	Yb mg/L	Lu mg/L	Sc mg/L	Th mg/L	U mg/L
CL1 composite	60	7.71	39	6.51	178	16.4	1.95	10.9	1.53	0.1	0.09	0.25

Oxalic Acid Precipitation Tests

A total of 5 tests were performed in order to find the optimum oxalic acid:REE stoichiometric ratio for the REE precipitation from heap leach solution; this ratio was calculated as the stoichiometric amount of oxalic acid needed to react with the REE present in solution. It should be noted that, even though there was a considerable amount of Al and other base metals in solution they were not taken into account for the oxalic acid calculation. Table 5-36 shows the main test conditions. The precipitation efficiencies for these tests are presented in Table 5-37.

Table 5-36 Oxalic Acid Precipitation - Test Conditions

Test ID	OX1	OX2	OX3	OX1A	OX2A
Feed Sample	CL1 composite	CL1 composite	CL1 composite	Ox1 PLS	Ox2 PLS
Feed, mL	200	200	200	148	154
Temp, °C	room	room	room	room	room
Time Mixing, min	60	60	60	60	60
$H_2C_2O_4$ Conc.	10%	10%	10%	10%	10%
stoich $H_2C_2O_4$:REE	72%	145%	217%	250%	300%

Table 5-37 Oxalic Acid Precipitation - Precipitation Efficiency (Reported as %)

Element	OX1	OX2	OX3	OX1A	OX2A
Si	0	0	0	0	0
Al	3	1	0	2	1
Fe	0	0	0	0	0
Mg	0	0	0	0	0
Ca	3	0	0	1	0
P	0	0	0	0	0
Mn	0	0	0	0	0
La	1	15	38	60	71
Ce	1	26	58	76	86
Pr	2	39	73	85	94
Nd	4	53	84	91	97
Sm	3	66	91	95	99
Eu	5	69	92	95	99
Gd	5	65	91	94	99
Tb	5	60	88	92	98
Dy	4	54	85	90	97
Ho	4	48	81	87	95
Y	2	31	65	76	87
Er	5	41	76	84	94
Tm	5	36	70	82	91
Yb	3	32	67	82	91
Lu	4	32	66	82	90
Sc	9	33	32	18	16
Th	0	61	60	66	53
U	3	0	0	2	3

All the tests were run at room temperature and for 60 minutes. The first 3 tests (OX1, OX2 and OX3) used CL1 composite as feed and 72%, 142% and 217% oxalic acid:REE ratio, respectively. REE precipitation in these tests were low, particularly in OX1 were none of the REE precipitated more than 5% and OX2 where the highest precipitation efficiencies were around 60%.

Based on the results from these tests, especially OX1 and OX2 (lower extractions) two other tests were run, OX1A and OX2A. Due to the low availability of CL1 composite solution, OX1 and OX2 final solutions were used as feed for OX1A and OX2A. Oxalic acid:REE ratio for these tests were 250% and 300%. REE precipitation increased with the increment of the oxalic acid:REE ratio, though even at its highest value (OX2A, 300%) not all REE were completely precipitated, more remarkable La 71% and Y 87%.

Oxalic acid was selective against the gangue material, Si, Al, Fe, Mg, Ca, P and Mn precipitation was mostly zero regardless of the oxalic acid added. Th precipitation was around 60% when oxalic acid addition was higher than 145%. Due to the small amount of solids produced in these tests solids assays were not possible to obtain therefore there is no information on REE precipitate quality.

SGS Lakefield Aluminium Removal Tests

A total of 3 tests were conducted in order to find a pH range in which aluminium would be precipitated while most of the REE remain in solution. Table 5-38 shows the main test conditions. The precipitation efficiencies for these tests are presented in Table 5-39.

Table 5-38 Aluminium Removal - Test Conditions

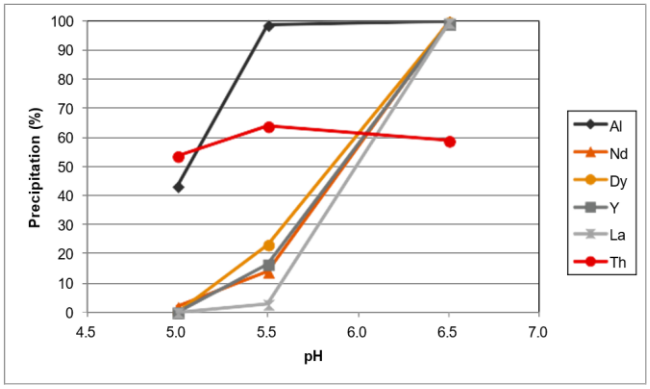
Test ID	AR1	AR2	AR3
Feed Sample	CL1	CL1	CL1
Feed, mL	200	200	200
Temp, °C	room	room	room
Time Mixing, min	60	60	60
NH_4HCO_3 Conc.	50 g/L	50 g/L	50 g/L
pH target	5.0	5.5	6.5

Table 5-39 Aluminium Removal - Precipitation Efficiency (Reported as %)

Element	AR1	AR2	AR3
Si	25	46	33
Al	43	99	100
Fe	0	0	0
Mg	0	0	0
Ca	0	2	8
P	0	0	0
Mn	0	0	0
La	0	3	99
Ce	0	5	100
Pr	0	8	100
Nd	2	14	100
Sm	0	15	100
Eu	1	19	99
Gd	0	15	100
Tb	2	21	100
Dy	0	23	100
Ho	1	24	99
Y	0	17	99
Er	1	29	99
Tm	2	35	97
Yb	0	45	99
Lu	0	45	98
Sc	17	24	14
Th	54	64	59
U	0	31	75

As is shown in Figure 5-22 when working at pH 5.0 aluminium removal was 40% while REE co- precipitation was below 2%. Worth noting is the fact that 50% of the Th was also removed at this pH. When pH was increased to 5.5 aluminium removal was 99%; REE co-precipitation also increased (14% Nd, 23% Dy and 17% Y) as a result of the higher pH. REE co-precipitation was >99% when the pH was increased to 6.5.

Figure 5-22 Aluminium Removal - Precipitation vs. pH



5.5.4.4 SGS Lakefield Conclusions and Recommendations

A series of scoping-level tests investigating the extraction of rare earth metals from weathered crust elution-deposited mineralization (REE Clay) samples were performed. The main conclusions and recommendations are as follows:

- Rare earth metals can be extracted from REE clay by ion desorption using an ammonium sulphate or sodium chloride solution as eluant. Main extractions were around 88% Nd, 73% Dy, 67% Y, 86% La. It was also confirmed that most of the gangue material as well as Th and U remain in the solids and do not follow the REE.
- It was determined that shaking tests with a single contact and three eluant washes was the optimum and most practical method to evaluate REE extraction from REE clay samples. This method of extraction was run for 60 minutes and at room temperature making it very simple and feasible for running a large number of samples. It is recommended to apply this method in future testing (such as proposed in the mineralization variability test program proposal– 14097-PR2).
- Optimum eluant and optimum test conditions were determined in the SGS Lakefield Optimum Eluant Tests. A solution of 1 mol/L ammonium sulphate at pH 4.0 produced the maximum REE extractions and still achieved low gangue material extractions. A higher concentration of eluant as well as adjusting the pH to 2.0 increased Th and U extractions. When using sodium chloride, a concentration of 1 mol/L and an adjusted pH of 4.0 were determined as optimum conditions for high REE extractions and low Th and U extractions.
- Different eluant:ore ratios were tested in an extraction isotherm style series of tests. The data showed that despite low eluant:ore ratios, high REE extractions can be obtained. The data shows that a simple counter current desorption process should be capable of producing high grade REE liquors while at the same time producing low residue levels (i.e. high extraction).
- Heap leaching was simulated in a series of small (scoping level) column leach tests. Two columns were operated for 218 hours; Column 1 was run using a solution of 1M ammonium sulphate at pH 4 as eluant while Column 2 ran with a solution of 1M ammonium sulphate at pH 4. The irrigation rates were 14.1 and 12.8 L/h/m^2 for Column 1 and 2, respectively. Maximum REE extractions were accomplished in Column 1 using ammonium sulphate (88% Nd, 73% Dy, 67% Y, 86% La). Column 2 (sodium chloride) led to lower extractions of 78% Nd, 68% Dy, 63% Y and 82% La. Not only were the extractions lower in Column 2 they also took more time to achieve those extractions as is shown in Figure 6. Gangue extractions as well as Th and U extractions remained low in Column 1 and Column 2.
- Based on the observed feed moisture difference and its (assumed) effect on the physical column behavior, it is recommended to run further column tests investigating the effect of agglomerate curing time. In addition it is recommended to conduct larger column tests and also to investigate the effect of heap leaching breccia or breccia/clay blends. It is also recommended to examine the merits of counter current heap leaching. Many of these recommendations are addressed in SGS proposal 14097-PR3.

- A series of Oxalic Acid Precipitation Tests were run with the solution collected during heap leaching (Column 1). Different oxalic acid:REE stoichiometric ratios were tested. REE precipitation increased with the increment of the oxalic acid:REE ratio, though even at its highest value (OX2A, 300%) not all REE were completely precipitated, more remarkable La (71%) and Y (87%).
- Oxalic acid was selective against the gangue material, Si, Al, Fe, Mg, Ca, P and Mn precipitation was mostly zero regardless of the amount of oxalic acid added. Thorium precipitation was around 60% when oxalic acid addition was higher than 145%. Due to the small amount of solids produced in these tests solids assays were not possible to obtain therefore there is no information on REE precipitate quality.
- A series of Aluminium removal tests were also carried out using solution collected during heap leaching (Column 1). The pH of the solution was adjusted using a solution of 50 g/L ammonium bicarbonate. When working at pH 5.0 aluminium removal was 40%, while REE co- precipitation was below 2%. Worth noticing is the fact that 50% of the Th was also removed at this pH. When pH was increased to 5.5 aluminium removal was 99% but REE co-precipitation also increased (14% Nd, 23% Dy and 17% Y). REE co-precipitation was >99% when the pH was increased to 6.5.
- A two-step aluminium removal seems to be the more practical approach, though re-leaching of co-precipitated REE should be studied. The Al-free solution could be subjected to oxalic acid precipitation were the dosage would much lower (since there would not be any aluminium present) or alternatively REE can be recovered by a further ammonium bicarbonate addition.

5.5.5 In-situ leaching

A 1km^2 test area was selected by Tantalus in 2014 for in-situ leaching but testing did not proceed in 2015. Fourteen drillholes were drilled with the goals of establishing a hydrogeological model of the watershed where the proposed pilot plant was to be located and investigating the aquifer in regard to availability and quantity of water to run said pilot plant.

It is noted that Harena have committed that the in-situ leaching option is not being considered for the Ampasindava Project.

5.5.6 Metallurgical Conclusions

The metallurgical performance of the regolith represents one of the primary risks of this project given the relatively low overall grade of the material. To date metallurgical tests have been undertaken on material that more often than not has higher grades than the average grade of the deposit, and on relatively small samples. Future testwork should be undertaken on a broader selection of sample types (Variability Study) and on larger scale tests.

In order to track the variability in the solubility of the REEs within the deposit, the authors recommend that a “solubility test” be included as part of any future exploration assay protocol. These tests could be undertaken at the assay lab as a matter of course. Incorporating such a procedure into the exploration program will provide valuable information as to the variability in the proportion of REEs in the deposit that are readily extractible, both across the lateral extent of the orebody, with depth, with the variable parent rock and grade of the material itself.

Due to the rare earth grade of the material, heap leaching is the most practical approach to valuable metal recovery. The two tests performed as described above indicate that an additional column leach program be undertaken that would more precisely define variables such as lixiviant concentration, irrigation rate, agglomeration methods, possible binder usage and pulse leaching.

5.6 Mineral Resources

The current Mineral Resource Estimate (“MRE”) for the Ampasidava Project was first derived and reported by SGS in 2014 with an original effective date of October 20th, 2014.

SGS was commissioned by Harena Resources to review and restate the MRE in accordance with the guidelines of the JORC 2012 Code in 2023. The restated MRE issued by SGS (Camus, 2023) summarised below has an Effective Date of November 1st, 2023.

5.6.1 Current Mineral Resource

5.6.1.1 Introduction

Modelling was completed in Genesis software and the geostatistics was completed in Geostat+. The methodologies used for modelling and grade interpolation were according to industry standards, with the development of methods specific for the geometry of the deposit in question. The extension of the deposit is approximately 20,000 m by 10,000 m and its thickness is usually varying between 1 and 10 metres. The most frequent method for resource modelling is by sectional interpretation to create a 3D model of the deposit. The flat geometry of the deposit required a different approach. The following steps were followed for the resource estimation:

- Reception of the data and visits
- Validation of the drill hole database
- Selection of the mineralised intervals for each drill hole for each layer (PED and SAP)
- Selection of the topography surface to be used for the model
- Creation of volume models of the layers in 2D (2D XY model including thickness and Z position)
- Variogram modeling in 2D for 19 variables for each layer
- Conversion in 3D block model to use in Genesis software (3D XYZ model with PED and SAP tags)
- Estimation of the grades for 19 variables for each layer in the 3D block model
- Classification of the resource according to drill hole spacing
- Creation of solids within barren areas to constrain the resource
- Validation of the density for each layer and for each prospect (Ambaliha, Ampasibitika, Ampasibitika South, Befitina, Caldera and North West Territories)
- Queries on the 3D block model but with cut-off grade applied on the average grade over the total thickness

5.6.1.2 Exploratory Data Analysis

SGS received the final “test pits and drill holes” database (DB) in the form of two Excel files on August 21, 2014. Additional assays (90) were received on September 9. Some updates came up to September 11 regarding trenches. Section 5.4.6 contains the complete details on the verification and changes that were made to the database prior to resource estimation.

Table 5-40 shows the variables in the database that were used in the resource estimation, while Table 5-41shows the conversion factors from ppm REE to ppm REO, together with the definitions of calculated estimates (TREO, MREO).

The author considers the data to be of sufficient quality to be used in a resource estimation.

Table 5-40 Variables used in the Resource Calculation

Element		
Number	**Abbreviation**	**Name**
1	Y	Yttrium
2	La	Lanthanum
3	Ce	Cerium
4	Pr	Praseodymium
5	Nd	Neodymium
6	Sm	Samarium
7	Eu	Europium
8	Gd	Gadolinium
9	Tb	Terbium
10	Dy	Dysprosium
11	Ho	Holmium
12	Er	Erbium
13	Tm	Thulium
14	Yb	Ytterbium
15	Lu	Lutetium
16	Nb	Niobium
17	Ta	Tantalum
18	Th	Thorium
19	U	Uranium

Table 5-41 Calculated Variables used in the MRE

Number	Calculated Variable	Units	Formula
1	Y_2O_3	ppm	Y/0.7874
2	La_2O_3	ppm	La/0.8527
3	Ce_2O_3	ppm	Ce/0.8538
4	Pr_2O_3	ppm	Pr/0.8545
5	Nd_2O_3	ppm	Nd/0.8574
6	Sm_2O_3	ppm	Sm/0.8624
7	Eu_2O_3	ppm	Eu/0.8636
8	Gd_2O_3	ppm	Gd/0.8676
9	Tb_2O_3	ppm	Tb/0.8688
10	Dy_2O_3	ppm	Dy/0.8713
11	Ho_2O_3	ppm	Ho/0.873
12	Er_2O_3	ppm	Er/0.8745
13	Tm_2O_3	ppm	Tm/0.8756
14	Yb2O3	ppm	Yb/0.8782
15	Lu2O3	ppm	Lu/0.8794
16	Nb2O5	ppm	Nb/0.6990
17	Ta2O5	ppm	Ta/0.8190
18	ThO2	ppm	Th/0.8788
19	U3O8	ppm	U/0.8480
20	TREO	ppm	$Y_2O_3+Eu_2O_3+Gd_2O_3+Tb_2O_3+Dy_2O_3+Ho_2O_3+Er_2O_3+Tm_2O_3+$ $Yb_2O_3+Lu_2O_{3+}La_2O_3+Ce_2O_3+Pr_2O_3+Nd_2O_3+Sm_2O$
21	MREO	ppm	Pr_2O_{3+} Nd_2O_{3+} Tb_2O_{3+} Dy_2O_3

5.6.1.3 Mineralised Intervals

The selection of mineralised intervals (MI) in the test pits and drill holes was executed solely by the lithologies. The first layer encountered from the surface is usually summarised as “PED” and the second layer as “SAP”. There are some intercalations where some SAP is contained in the PED layer and the opposite. Sometimes some “PED-SAP” material is between the “PED” and the “SAP”. A few other rock types are encountered inside the “PED” and “SAP” layers. Because the “PED-SAP” chemistry is similar to the “SAP” chemistry, all “PED-SAP” summaries were changed to SAP for the MI creation. When “MDS”, “SYE”, “OTH” or “SST” were found in-between “PED” and “SAP” layers, it was included within “PED” (based on average chemistry). When “FAS” or “MDS-FAS” were found in- between “PED” and “SAP” layers, it was included within “SAP” (again based on average chemistry).

It was discovered that there are two holes in the same location in 16 instances and three holes in the same location in 3 instances. Each time, one of the “PED” mineralised intervals was selected randomly and one of the “SAP” mineralised interval was selected randomly. In total, 21 “PED” and 19 “SAP” mineralised intervals were removed from the estimation.

The count and lengths of the final mineralised intervals, along with the contained assay statistics, and the contained gaps (lengths not assayed) are summarised in the Table 5-42. The Figure 5-23 shows some test pits and drill holes on section centered at 8,469,900 mN with actual proportions. The Figure 5-24 shows the same section with compressed horizontal distances (equivalent to a vertical exaggeration of 10).

Table 5-42 Count and Lengths of Mineralised Intervals, Contained Assays, and Gaps

	Mineralised Intervals		Assays		Gaps	
	Count	Length	Count	Length	Length	%
PED	4,369	13,781	13,926	13,775	6.2	0.04%
SAP	3,730	15,163	16,133	15,158	5.65	0.04%
TOTAL	8,099	28,944	30,059	28,932	11.85	0.04%

Figure 5-23 Normal View of Test pits and drill holes with Mineralised Intervals (MIs)

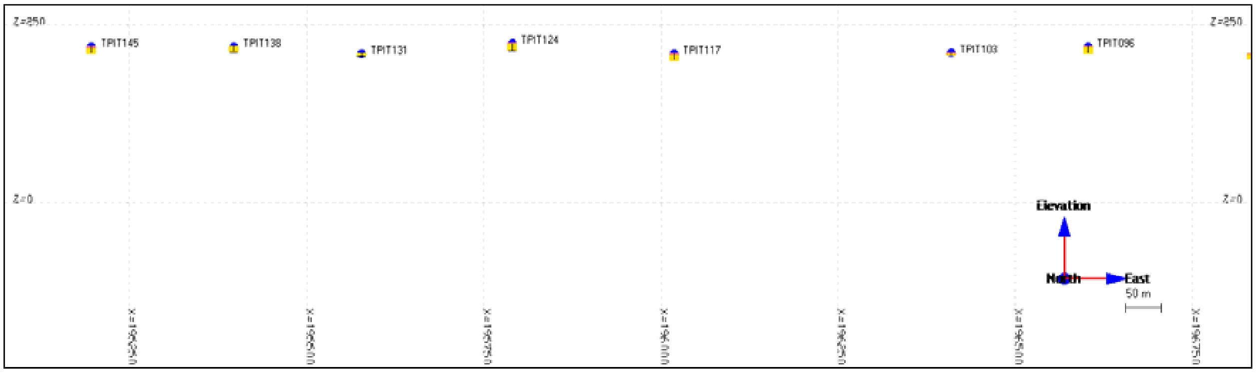
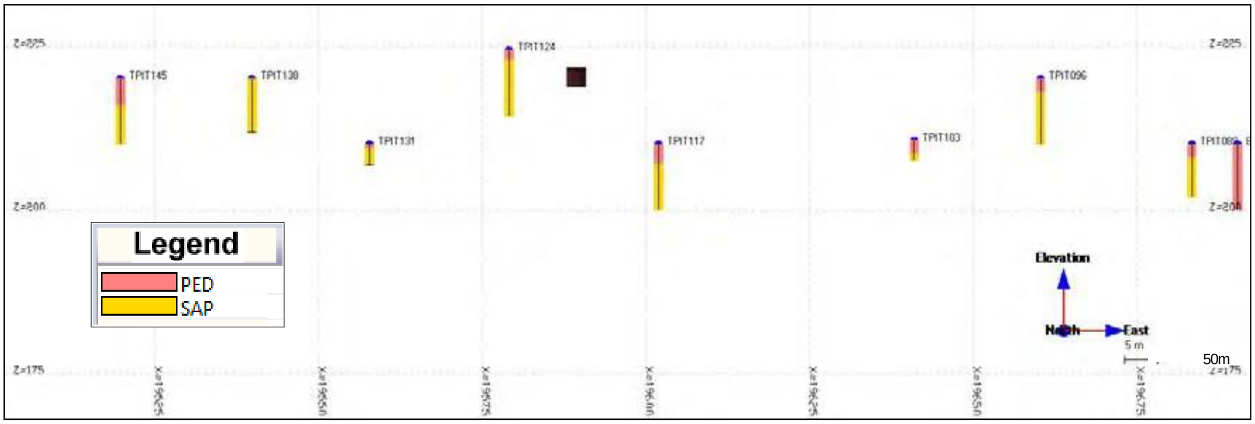


Figure 5-24 View of Test pits and drill holes with Mineralised Intervals shown with a 10x vertical exaggeration



5.6.1.4 Density

A total of 4569 dry densities measurements were received by SGS. The data was validated by plotting the dry density vs the size of the samples, and the dry density vs the wet density. A total of 4,309 dry densities measurements and 4,126 wet density measurements were selected as valid data for density determination. Table 5-43 shows statistics on retained density measurements and Table 5-44 shows the dry densities chosen for the resource estimation. Analysis was undertaken to determine if the density of the “PED” and “SAP” layers were discrete populations and to determine if the density was related to the grade. No correlation was found in either case. There is variability in the density and the water content but only the geographic location seems to be a determining factor.

Table 5-43 Statistics on Wet and Dry Densities and Water Contents for Each Prospect

Prospect	Wet Density					Dry Density					Water Content				
	Count	P25	Median	P75	Mean	Count	P25	Median	P75	Mean	Count	P25	Median	P75	Mean
Ambaliha	862	1.46	1.55	1.65	1.56	-	NA	NA	NA	NA	-	NA	NA	NA	NA
Ampasibitika	852	1.47	1.60	1.73	1.60	206	1.02	1.10	1.20	1.11	198	33%	40%	48%	40%
Ampasibitika S.	5,024	1.46	1.60	1.72	1.58	751	1.04	1.16	1.29	1.16	703	26%	33%	47%	36%
Befitina	2,342	1.36	1.50	1.60	1.48	-	NA	NA	NA	NA	-	NA	NA	NA	NA
Caldera	10,604	1.37	1.50	1.64	1.50	3,333	1.00	1.10	1.20	1.11	3,209	26%	34%	44%	36%
North West T.	1,429	1.38	1.52	1.64	1.51	19	1.04	1.10	1.20	1.11	16	22%	32%	38%	32%
TOTAL	**21,113**	**1.40**	**1.53**	**1.66**	**1.53**	**4,309**	**1.01**	**1.11**	**1.21**	**1.12**	**4,126**	**26%**	**35%**	**45%**	**36%**

Table 5-44 Densities Used for the Resource Estimation

Prospect	Chosen Density for Resources (t/m^3)
Ambaliha	1.10
Ampasibitika	1.10
Ampasibitika South	1.15
Befitina	1.10
Caldera	1.10
North West Territories	1.10

5.6.1.5 Topography Surface

Five separate topographic surfaces were supplied to SGS for consideration for the project.

1. Collars elevation data: covers all the project area, was taken by consumer grade handheld GPS. High accuracy but too low precision for use
2. Government map topography (contours): covers more than the project area. Contours every 25m. Medium accuracy (after SGS custom corrections) and medium-low precision
3. SRTM topography: elevation points every 90m provided by the NASA, covers more than the project area. Medium-low accuracy (after SGS custom corrections) and medium-low precision
4. Fugro geophysical survey elevation information: covers 17% of the project area. Medium-high accuracy (after SGS custom corrections) and medium-high precision
5. Digitised topo map (10m contours apparently from Government maps and Fugro combined), covering the entire project area. It contained contours every 10 m, with a medium-high accuracy and medium precision.

5.6.1.6 Volume Modeling of the Layers in 2D

Since this deposit is mainly a bi-dimensional structure, it made sense to produce a 2D model with thicknesses for the volume modeling. Since we will want to produce a mine design using 3D software, we will have to convert it to a 3D model.

Each of the test pits and drill hole locations has a thickness for both the “PED” and the “SAP” layers. We have drawn a tight outline of the drilled areas and forced the “PED” and the “SAP” to zero there. Any thickness of zero was instead forced to -1m to allow for a buffer zone around it. The thicknesses were modeled using one tin surface for the “PED” and another tin for the “SAP”.

5.6.1.7 Conversion of the 2D Volume Modeling into a 3D Block Model

The grid chosen for the 3D block model (BM) is 30 m x 30 m x 1 m. A 2D grid of points with the same spacing as the 3D BM was drawn (for each block center) and points at all sounding coordinates were added. These points were projected on the 2 tin surfaces representing the thickness for the “PED” and “SAP” layers. These points were also projected on the selected topography surface. From the topography elevation of those points and the thicknesses of the layers (forced to zero when negative), the bottom surfaces of the “PED” and the “SAP” layers were designed using simple triangulation (tin – displayed in Figure 5-27) and the 3D blocks pertaining to each layer were listed (with a 1 m thickness). The coordinates and size of the blocks are listed in Table 5-45.

Figure 5-25 shows the location of a representative cross-section of the test pits and regolith layers on the project scale, while Figure 5-26 shows a close-up plan of the area and Figure 5-27 is the representative section on 8.470,710N. An oblique section showing Block Model TREO values and deposit contours is presented in Figure 5-28

Table 5-45 Block Model Size and Coordinates

Direction	Size of blocks (m)	Coordinate of the center of the first block (m)	Number of blocks	Coordinate of the center of the last block (m)
X (East)	30	181,800	611	200,100
Y (North)	30	8,468,100	461	8,481,900
Z (Elevation)	1	-9.5	760	749.5

Figure 5-25 Project-scale view of location of representative cross-section

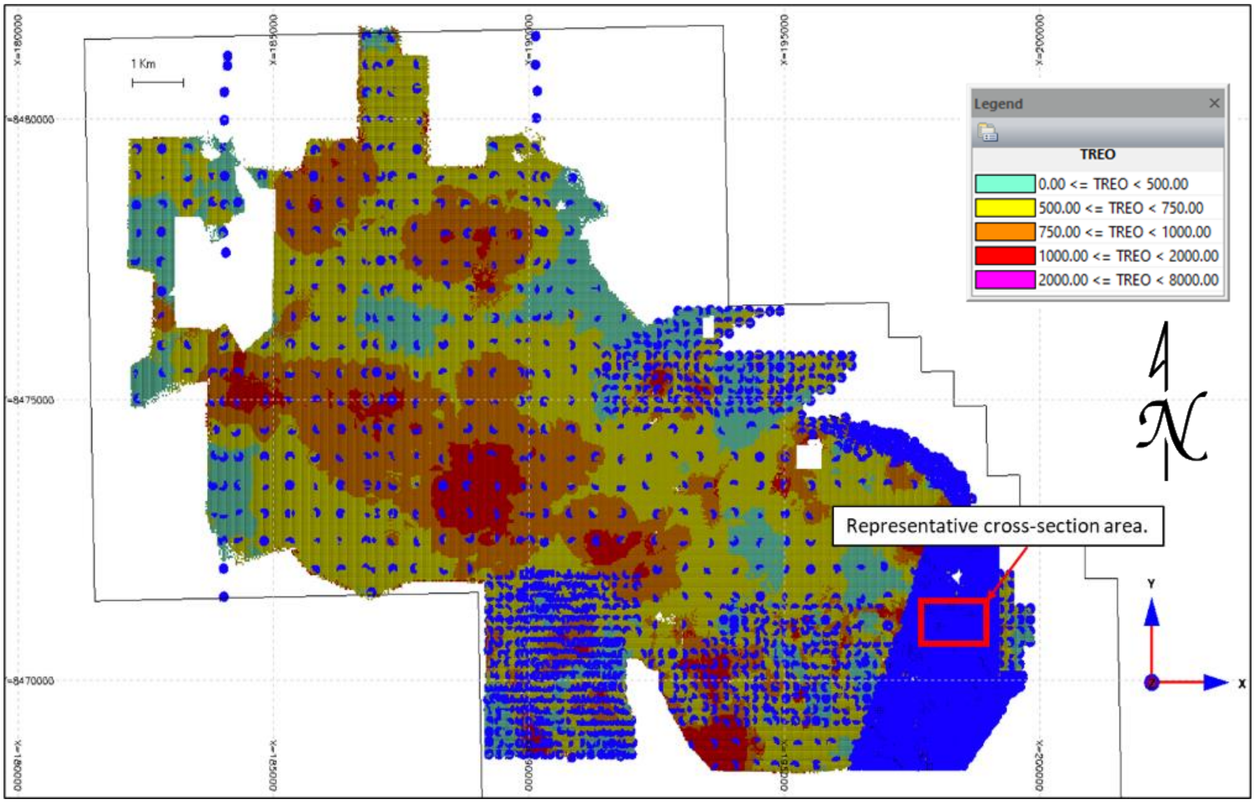


Figure 5-26 Close-up plan view of cross-section location

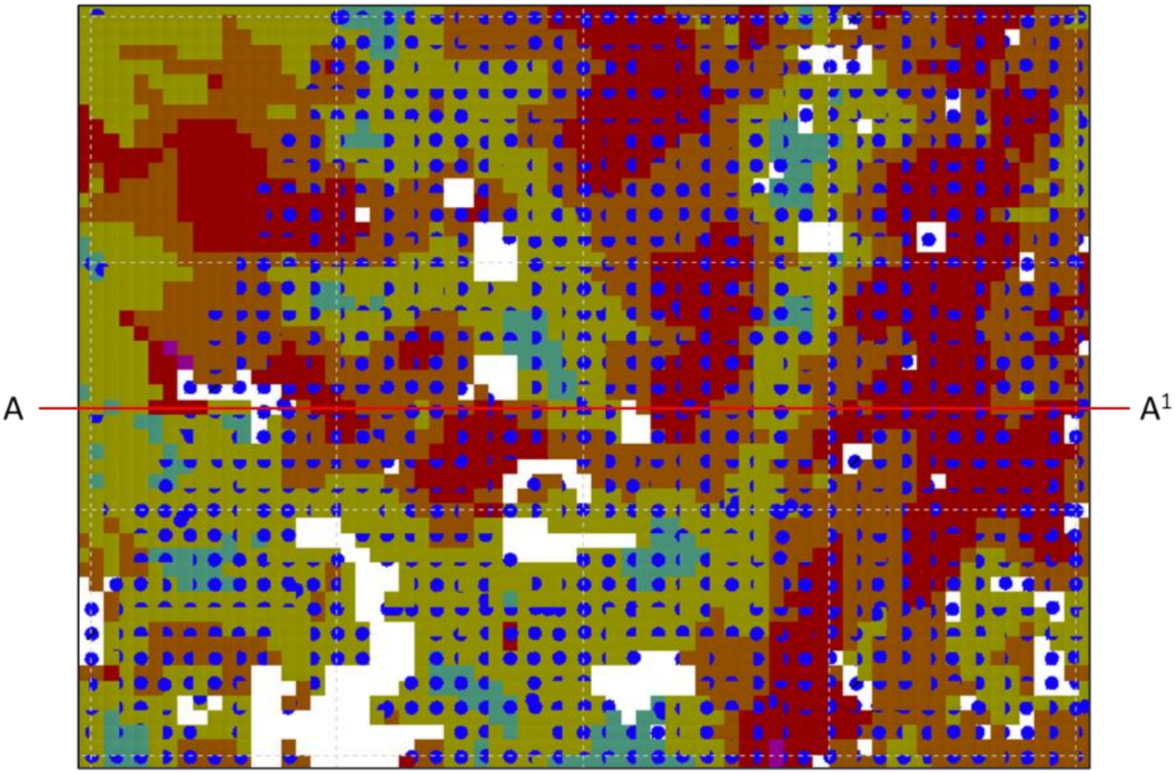


Figure 5-27 Representative Section on 8,470,710N Showing Test Pits and Pedolith (Blue) and Saprolite (Red) Horizons

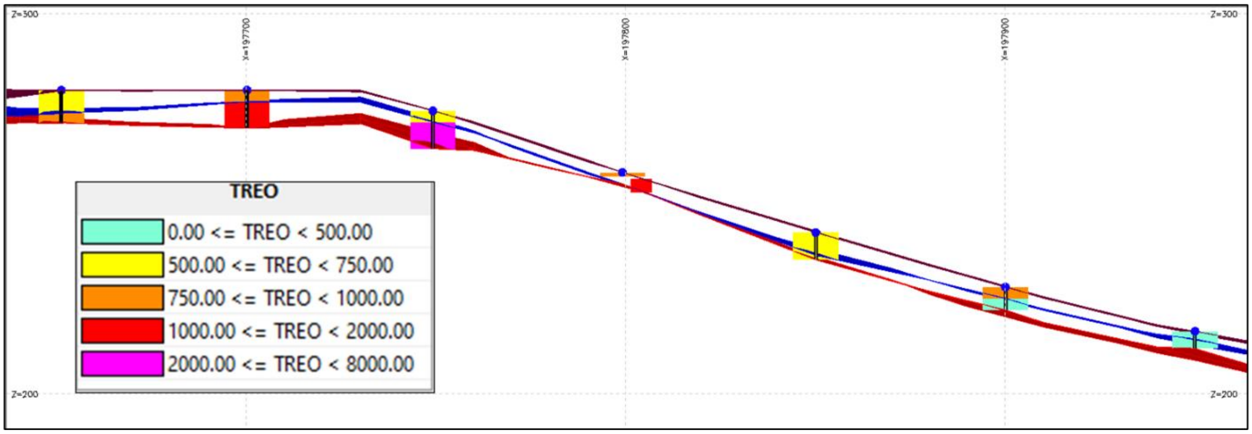
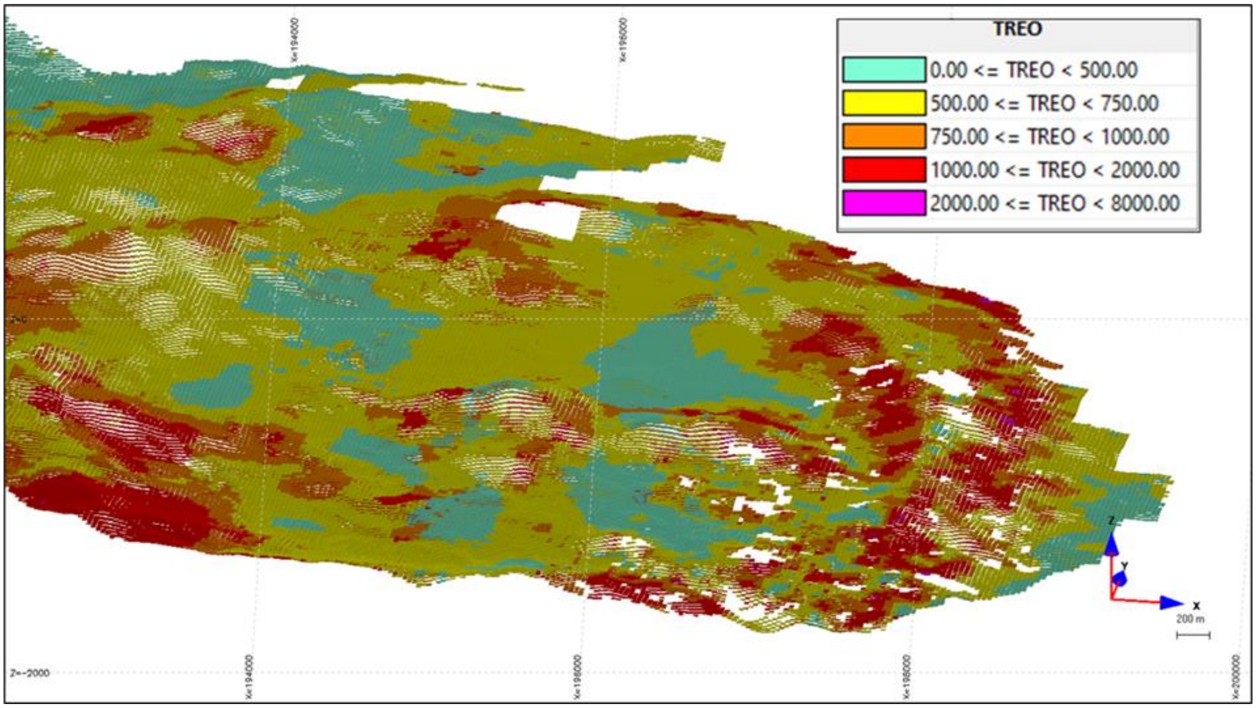


Figure 5-28 Oblique Section Showing Block Model TREO Values and Deposit Contours



5.6.1.8 High Grade Capping

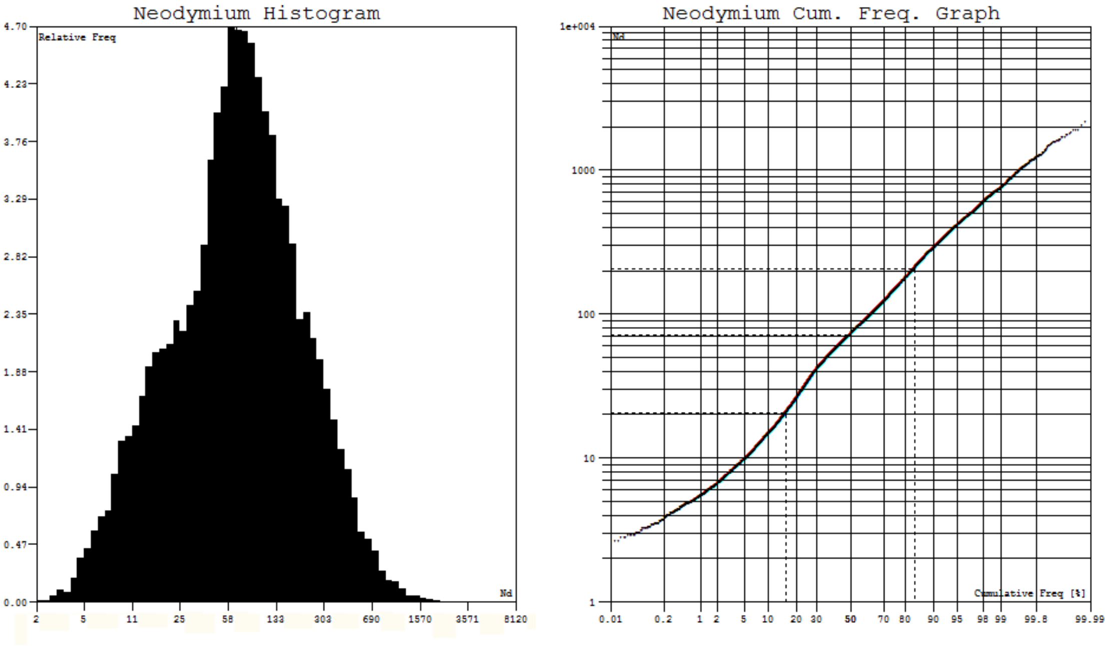
Out of 30,059 assay intervals inside the “PED” and “SAP” layers, most of them have a length at exactly 1 m (79% of them) and most of them have a length over 0.5 m (97.5 % of them). For convenience and to remove possible outliers, assay intervals shorter than 0.5 m were removed for the following capping study exercise. The shorter assays were assessed separately to make sure the conclusions on the requirement of capping was still sound.

To determine whether capping was required, the “metal contribution” of the highest 1% of the assay intervals was analysed. For example, the “metal contribution” of the 292 highest assays (out of 29,236) for Neodymium is 9%. Both the histogram and the cumulative frequency graph (Figure 5-29) on log scales do not indicate that Neodymium requires capping. All “metal contributions” of the 1% highest assays for all estimated elements are less than 10%. For all estimated elements, a capping of the 20 highest values has no significant effect on the contribution of the 292 highest assays.

While the populations are clearly log- normal distributions, there are no “nugget” values that have a significant impact on the metal content.

It is the opinion of the Competent Person that no high-grade capping was required for any of the elements modelled in the MRE.

Figure 5-29 Histogram and Cumulative Frequency (Nd Assays – “PED” and “SAP” Layers)



5.6.1.9 Modeling of Variograms

Variography on the composite data was undertaken to determine if viable variograms for the 19 variables could be created and if the data was amenable to estimation via kriging.

Variograms were created for variables in both “PED” and “SAP” horizons, with differences observed for the same variable in the different horizons. As such, 38 variograms were initially created and reviewed. During the review process, the 38 variograms were reduced to 11 variograms because many of them exhibited similar properties. The variograms were created in 2D using composites averaging the total thickness of the layer. The elevations were forced to zero to eliminate the 3D component of the variography since the deposit is essentially bi-dimensional. The list of retained variograms is in Table 5-46.

Table 5-46 List of Variograms Used for the Estimates

Layer	Elements	Nugget	Exponential 1		Exponential 2	
		Sill	Sill	Range (m)	Sill	Range (m)
PED	Y La Gd Dy Ho Er Tm Yb Lu	0.63	0.11	60	0.26	200
PED	Pr Nd Sm Eu	0.32	0.40	23	0.28	220
PED	Tb	0.55	0.25	30	0.20	230
PED	Ce	0.00	0.45	6.5	0.55	130
PED	Nb Ta Th	0.485	0.515	190		
PED	U	0.55	0.45	150		
SAP	Y Pr Nd Sm Gd Tb Dy Ho Er Tm Yb Lu	0.50	0.35	23	0.15	170
SAP	Eu	0.34	0.51	26	0.15	350
SAP	La Ce	0.00	0.825	9	0.175	150
SAP	Nb Ta	0.00	0.79	14	0.21	220
SAP	Th U	0.00	0.92	11	0.08	100

5.6.1.10 Block Model Interpolation

The variography results indicated that the mineralisation was amenable to estimation by kriging.

The kriging was completed in 3 dimensions and in 3 passes using the variogram data shown in Table 5-46.

A block discretization of 4 (X) x 4 (Y) x 1 (Z) was used for the estimation of the distance between a block and a composite. The “PED” and “SAP” layer blocks were estimated using the composites averaging the grades over the complete thickness of respectively the “PED” and “SAP” layers.

For the first pass, a search ellipsoid with radiuses of 90 m (X) x 90 m (Y) x 180 m (Z) was used with a minimum of 6 composites and a maximum of 9 composites.

For the second pass, a search ellipsoid with radiuses of 350 m (X) x 350 m (Y) x 700 m (Z) was used with a minimum of 6 composites and a maximum of 9 composites.

For the third and last pass, a search ellipsoid with radiuses of 900 m (X) x 900 m (Y) x 1800 m (Z) was used with a minimum of 3 composites and a maximum of 9 composites.

Because each hole has only one composite per layer, no “maximum composites per drillhole” was set. In addition, no octant search was used.

With the settings described above, a total of 2% of the blocks were not estimated because less than 3 test pits and drill holes were available in a radius of 900 m.

5.6.1.11 Mineral Resource Classification

This MRE for the Ampasindava Project deposit is prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code).

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Inferred Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.

Measured Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation where data and samples are gathered.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Ore Reserve or under certain circumstances to a Probable Ore Reserve.

Classification of the Ampasindava Project

The classification of the Ampasindava Project was accomplished by drawing outlines of areas that have been drilled to a hole/pit grid spacing.

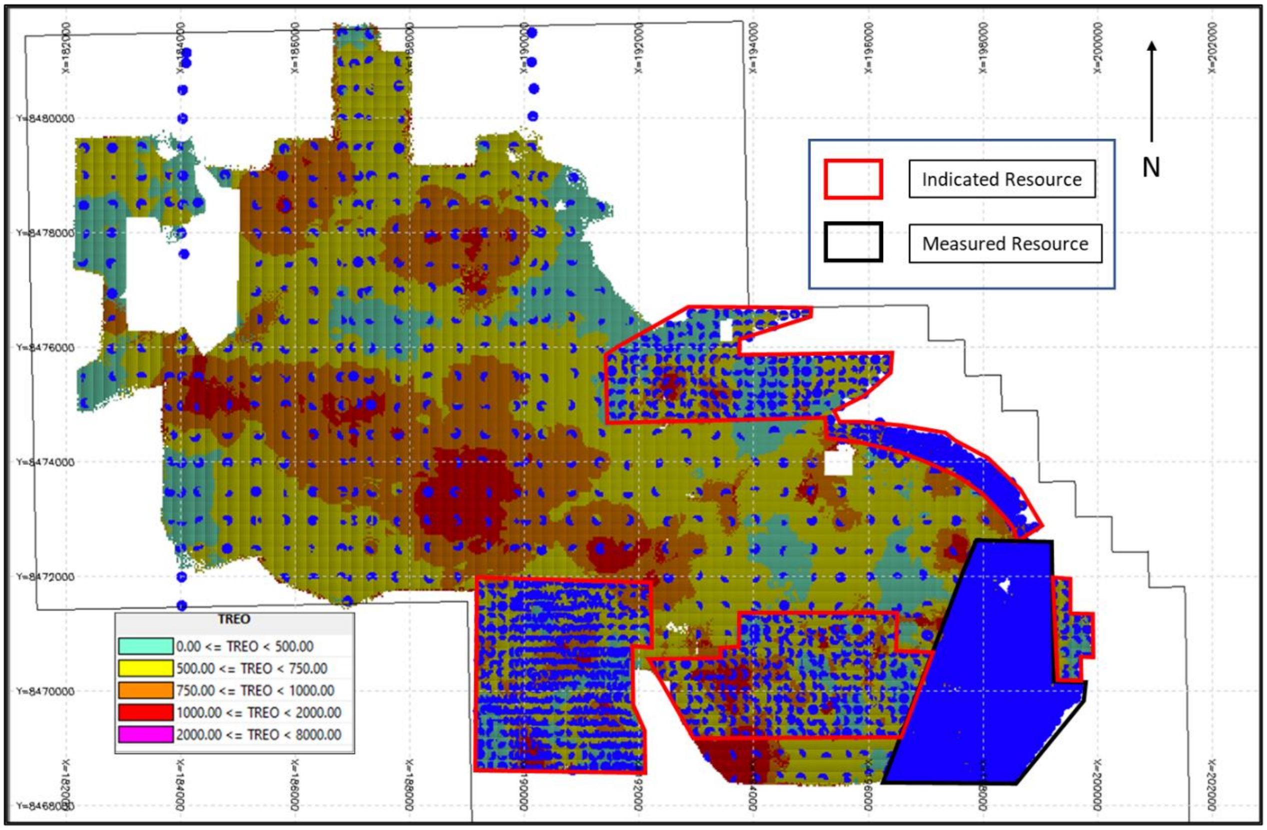
- A grid spacing of at least 50 m x 50 m corresponds to measured resources.
- A grid spacing of at least 200 m x 200 m corresponds to indicated resources.
- A grid spacing of at least 500 m x 500 m corresponds to inferred resources.

Figure 5-30 shows the outlines for the different classifications of material along with the collars of the test pits and drill holes.

Some of the test pits are in close proximity to the property boundary. No test pits and drill holes are outside of the property and no resources were modelled outside the property.

It was noted that some areas are missing test pits and drill holes to make a complete grid. It is possible that the test holes were not completed because there was no “PED” or “SAP” material in that area. These areas were identified in the MRE and were excluded from the resource classification.

Figure 5-30 Indicated and Measured Classification with Drillhole and Test Pit Locations



5.6.1.12 Cut-Off Grade Determinations

To determine the base case cut-off grade (COG), the first steps were to determine the list of prices to be used for the evaluation of the reasonable prospect for economic extraction. Then, the processing recoveries and the marginal costs were estimated. A discount of 27% was applied to the in-situ value to account for last processing stage (REO separation).

The basket price of the REOs was supplied by Harena and is shown in Table 5-47.

The cut-off grade calculations are based on the deposit being processed via heap leach technology. Some recoveries were determined through metallurgical test work, but a full set of recovery data was not available. As such, assumed recoveries were applied to determine the cut-off grade.

Table 5-48 shows the calculations used to convert REE to REO, together with the assumed recoveries and recovered oxide concentrations from the deposit.

Table 5-47 List of the Prices Used for REOs for the 2023 MRE

Rare Earth Oxide	Price (US$/kg)
Ce_2O_3	1.45
La_2O_3	1.4
Pr_2O_3	176
Nd_2O_3	185
Sm_2O_3	4.63
Eu_2O_3	33.1
Gd_2O_3	102
Tb_2O_3	2220
Dy_2O_3	509
Ho_2O_3	292
Er_2O_3	64.2
Tm_2O_3	850
Yb_2O_3	17.9
Lu_2O_3	866
Y_2O_3	15

Table 5-48 Conversion Factors and Recoveries for REOs

Conversion	Metal (REE)		Oxide (REO)			Weight of	Processing	Weight of
Factors (REE/REO)	Name	Grade (ppm)	Name	Grade (ppm)	Price List ($/kg)	In-Situ Oxide (kg/t)	Recovery	Recovered Oxide (kg/t processed)
0.7874	Y	72	Y2O3	91	15	0.091	59%	0.05
0.8527	La	176	La2O3	206	1.4	0.206	82%	0.17
0.8538	Ce	272	Ce2O3	319	1.45	0.319	12%	0.037
0.8545	Pr	34	Pr2O3	39	176	0.039	82%	0.032
0.8574	Nd	112	Nd2O3	131	185	0.131	81%	0.11
0.8624	Sm	19	Sm2O3	22	4.63	0.022	76%	0.016
0.8636	Eu	2.1	Eu2O3	2.4	33.1	0.0024	73%	0.002
0.8676	Gd	15	Gd2O3	17	102	0.017	73%	0.013
0.8688	Tb	2.2	Tb2O3	2.6	2220	0.0026	69%	0.002
0.8713	Dy	13	Dy2O3	15	509	0.015	63%	0.009
0.873	Ho	2.6	Ho2O3	2.9	292	0.0029	58%	0.0017
0.8745	Er	7.6	Er2O3	8.6	64.2	0.009	52%	0.004
0.8756	Tm	1.1	Tm2O3	1.2	850	0.0012	44%	0.0006
0.8782	Yb	7.1	Yb2O3	8.1	17.9	0.008	43%	0.004
0.8794	Lu	1.1	Lu2O3	1.2	866	0.0012	38%	0.0005
NA	TREE	736	TREO	868	60.16*	0.87	52%	0.45

*: 60.16 $ is the TREO basket price for the project

Table 5-49 outlines the parameters used to determine the final base-case cut-off grade for the Ampasindava Project.

Table 5-49 Parameters Used for Determination of Base-Case Cut-Off Grade

	Item	Unit	Value	Comments
Com-modity	**TREO Basket Price**	US$/kg	43.915	27% Discount Applied for REO Separation
	Unit Value (UV)	US$/Grade of Unit Recovered	0.044	
	Mining Recovery	%	95	
	Mining Dilution	%	5	
	Processing Recovery	%	51.8	
	Commodity Factor (CF)	US$/Grade of In-Situ Unit	0.023	Calculated Variable: Commodity = Grade*CF*Percent of Block within Wireframe
Costs	**Processing Cost (PC)**	US$/tonne	8	Heap Leach Processing
	G&A	US$/tonne	0.75	
	Mining Cost	US$/tonne	1.4	
	(PC+G&A)*(1+Dilu-tion)	US$/tonne In-Situ	9.1875	Calculated Processing Cost in Genesis
COG	**Calculated Cut-Off Grade**	ppm	492.9	
	Resource Cut-Off Grade	ppm	500	

5.6.1.13 Reasonable Prospects of Eventual Economic Extraction

The general requirement that all mineral resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, the REE mineralisation at the Ampasindava Project is considered amenable to strip mining extraction.

To determine the quantity of material representing “reasonable prospects for eventual economic extraction” by strip mining, economic optimisation parameters were applied to the block model and only those blocks which fall within these criteria are reposted in the MRE.

Table 5-49 shows the parameters applied to the block model to fulfil the “reasonable prospects for eventual economic extraction” criteria.

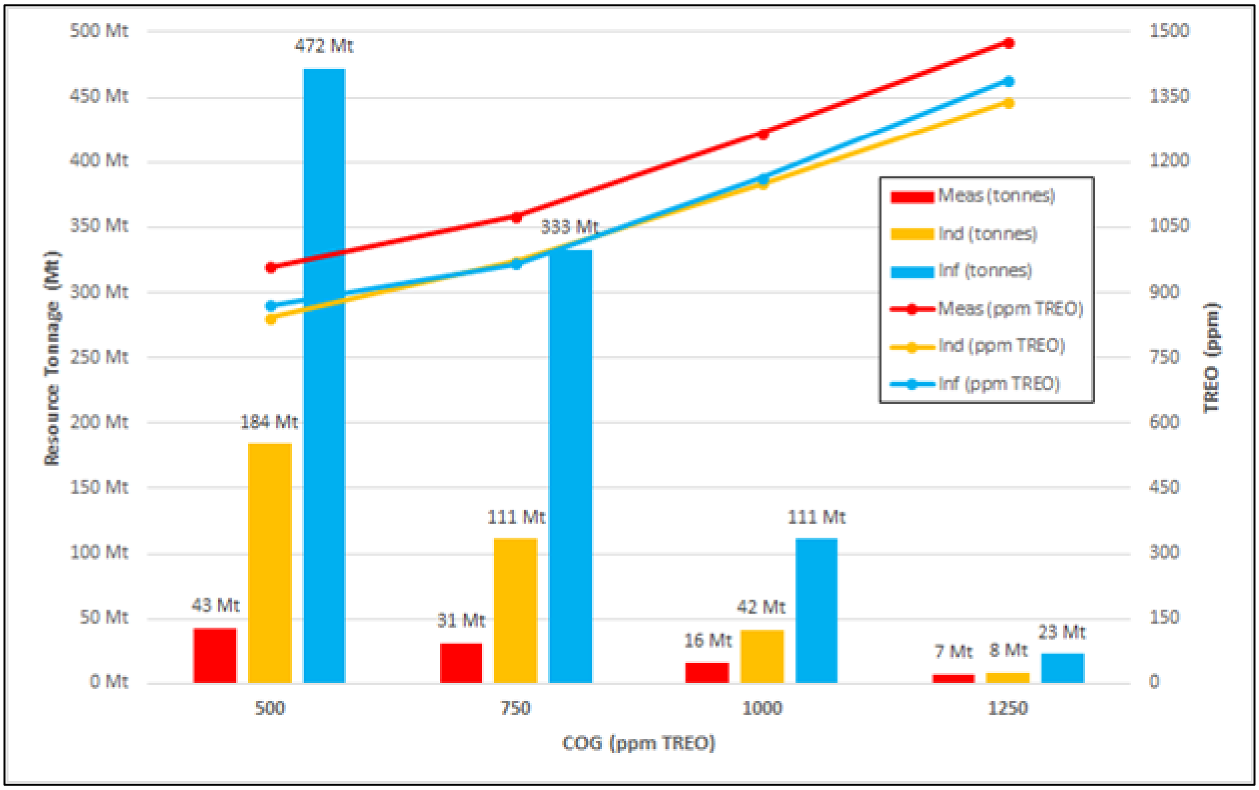
The reader is cautioned that the results from the optimisation parameters are used solely for the purpose of testing the “reasonable prospects for economic extraction” and do not represent an attempt to estimate mineral reserves. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

5.6.1.14 Sensitivity to Cut-Off Grade

The Ampasindava Project mineral resource has been estimated at a range of cut-off grades to demonstrate the sensitivity of the resource to cut-off grades. The current mineral resources are reported at a cut-off grade of 500 ppm TREO (Table 5-50).

Figure 5-31 shows the sensitivity to a 750 ppm TREO cut-off (+50% base case), a 1000 ppm TREO cut-off (+100% base case) and a 1,250 ppm TREO cut-off (+150% base case).

Figure 5-31 Ampasindava Project MRE Sensitivity to Cut-Off Grade



5.6.1.15 <u>Mineral Resource Statement</u>

The Mineral Resource Estimate is reported in Table 5-50 using a cut-off grade of 500 ppm TREO for the resource. The mineral resources are constrained by the topography and based on the conceptual economic parameters detailed in Table 5-49. The estimate has an effective date of the 1st November, 2023. The Competent Person for the estimate is Yann Camus, P.Eng., an SGS employee.

The MRE presents the results as Total Rare Earth Oxides (TREO), with Magnet Rare Earth Oxides (MREO) broken out. Definitions of TREO and MREO are in the notes to Table 5-50.

The restated Ampasindava MRE is presented in Table 5-50.

Table 5-50 Mineral Resource Estimate for Ampasindava Deposit at Cut-Off of 500 ppm TREO

Classification	Tonnage (Mt)	Volume (Mm³)	Area (Mm²)	Density (t/m³)	Thickness (m)			TREO (ppm)	MREO (ppm)	MREO / TREO ratio	Contained TREO (t)	Contained MREO (t)
					Total	PED	SAP					
Measured	42.5	38.1	7.0	1.11	5.46	2.85	2.60	958	221	23%	40,700	9,400
Indicated	184.0	167.1	25.0	1.10	6.70	2.65	4.04	842	178	21%	154,800	32,700
Measured + Indicated	**226.5**	**205.3**	**31.9**	**1.10**	**6.43**	**2.70**	**3.73**	**863**	**186**	**22%**	**195,500**	**42,100**
Inferred	472.0	429.1	78.9	1.10	5.44	2.71	2.73	870	189	22%	410,500	89,000
Total	698.5	634.3	110.8	1.10	5.72	2.71	3.02	868	188	22%	606,000	131,100

1. *The Mineral Resource Estimate (MRE) has an effective date of the 1st November, 2023. The Competent Person for the MRE is Mr. Yann Camus, P.Eng., an employee of SGS.*
2. *The classification of the current Mineral Resource Estimate is consistent with the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code).*
3. *All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.*
4. *All Resources are presented undiluted and in situ, constrained within a 3D model, and are considered to have reasonable prospects for eventual economic extraction.*
5. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
6. *Bulk density values were determined based on physical test work from each part of the deposit.*
7. *The base cut-off grade of 500ppm TREO considers a mining cost of US$1.40/t mined, a processing cost of $8.00/t mined and G&A cost of US$0.75/t mined.*
8. *TREO =* $Y_2O_3+Eu_2O_3+Gd_2O_3+Tb_2O_3+Dy_2O_3+Ho_2O_3+Er_2O_3+Tm_2O_3+Yb_2O_3+Lu_2O_3+La_2O_3+Ce_2O_3+Pr_2O_3+Nd_2O_3+Sm_2O_3$
9. *MREO =* $Pr_2O_3+Nd_2O_3+Tb_2O_3+Dy_2O_3$
10. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

5.7 Exploration Potential

The thickness of the regolith profile is fundamentally important given that this affects the volume of material available to host REE mineralisation. Erosion and truncation of the regolith profile due to drainage and steep slope gradients obviously have a detrimental effect on the completeness of the regolith profile. Therefore, geomorphological studies using aerial photography, satellite imagery and field mapping will facilitate the identification of areas favouring the accumulation of regolith material.

Geochemically the primary REE-enriched source rocks are distinct because of the comparatively unique combination of elements they contain. The apparent enrichment of Ce in the upper sections of the regolith profile (particularly the ferruginous zone) has already been considered as part of the soil sampling programs completed to date and may be utilised to identify more localised and higher concentrations of regolith-hosted REE mineralisation.

Whilst mineralised regolith material is not geophysically distinct, some of the REE-enriched source rocks are due to the presence of uranium- and thorium-bearing accessory minerals. This characteristic could be further utilised using the available airborne and ground radiometric data to help identify particularly favourable source rocks (where they are not overly concealed).

A related side, the overall uranium and thorium content of the regolith material (based upon the available geochemical results from pits, window sampling holes and core drillholes) is relatively low and averages just 12 ppm U_3O_8 and 57 ppm ThO_2. These concentrations are not considered to pose any environmental or anthropogenic risks. Particularly as these elements are not concentrated within the proposed solutions used for extraction.

It is anticipated that clay (and other) minerals in the regolith profile can be identified and discriminated using infrared spectrometry. Given that clay type and ionic exchange capacity are related properties, this method may provide an effective means of mapping the ionic character of the regolith profile.

Several sources have described the colour of the regolith as an important guide to mineralisation. This has not yet been established in the Project area, but it stands to reason that paler-coloured zones are more clay-rich than orange to red-coloured zones that contain more iron.

There is an opportunity on the Ampasindava deposit to extend known mineralization at laterally, and to identify additional zones of ionic-clay hosted REE mineralisation elsewhere on the Ampasindava Property. In addition, continued infill drilling of the Ampasindava deposit will help define with more precision the shape of the deposit, confirm the geological and grade continuity, and increase the resource confidence level (Inferred to Indicated or Measured, or Indicated to Measured). Harena Resources intentions are to direct their development efforts towards trial mining and advanced economic studies in 2023-2024, with exploration and resource growth not expected to be a primary focus on the Ampasindava Project in the immediate future.

5.8 Environmental Studies, Permitting and Community Impact

5.8.1 Environmental Reporting

SGS has consulted seven environmental monitoring reports (RSE: Rapport de Suivi Environnemental) covering the years 2009 to 2015 and two Annual Activity Reports for 2014 and 2015. These reports were completed by Tantalus in order to be compliant to the regulation of the Office National de l'Environnement (ONE) and the ministry of Mines of Madagascar.

The RSE 1 (covering period from November 2009 to November 2010) concluded that Tantalus had respected the terms of the Environmental and Social requirements for the first year of their Environmental Permit. Oil was stored on a waterproofed concrete floor to avoid the risk of contamination of the natural soil which could then contaminate groundwater. The RSE 1 reveals that the water used for drilling came from the river near the drilling and presents tables of water quantity taken from each river.

The RSE 2 (covering period from November 2010 to November 2011) had the same conclusion of the previous reports, adding some information about the rehabilitation of the road linking National Road 6 (Ambondromamy - Antsiranana) at Belintagno the main camp of Tantalus Ankatafa and the development of drilling platforms and open access to footpaths. The final location of boreholes and digging of wells was chosen to avoid or minimise damage to the environment (logging, land clearing). Nevertheless, the excavations could not be avoided especially during the construction of drilling platforms. Pits and trenches were backfilled with the excavated material also identified by their digging while taking into account the initial natural stratification facies soil. After filling, the ground is shaped to restore their original morphological configuration. Backfilled and the soil profiles were grounded seedlings for their re-vegetation.

From the point of view of environmental considerations, no sensitive area has been established within the exploration license for the 2009-2011 periods. However, the northeastern part of that exploration license, and an area of approximately 100 km^2, is located in a priority area for the establishment of protected areas.

The RSE 3 (covering the period from November 2011 to November 2012) reveals that for various reasons, Tantalus had not conducted any field work program during 2012. Following to a site visit done by ONE in December 2013 in order to verify the respect of the environmental regulations, some rectifications were outlined by Tantalus. The RSE 3 was conducted in a concise way to answer to each article of environmental requirements.

A letter sent by Tantalus at the INSTN (Institut National des Sciences Techniques et Nucléaires) on January 7, 2014 requesting a service to achieve adequate measures in order to define the radioactive background of the entire perimeter of mining research to possibly set a radioactive zoning. This request is made following a request by the environmental authorities (ONE).

The principal camp is located in the Ankatafa village, supplied with electricity from a generator and potable water. Noise reduction of the generator is achieved through a fitted soundproofing cover.

No handling of fuel is made on site. Vehicle maintenance taking place at gas stations in Antananarivo, the fuel supply to the site is at the gas Galana Ambanja where fuel is directly poured into the tanks of the vehicles. Biodegradable waste is buried at an approved site near the camps. The few non- biodegradable waste camps are transported to Ambanja for disposal at the city landfill.

Labor is recruited from surrounding villages and hamlets as much as possible. In case of need, it was appealed to the more remote villages and hamlets closer to Ambanja and sometimes more distant locations. In the early work, the framing of Tantalus agents located places and sacred sites or exercise of rites and burial sites from the villagers. At the start of drilling, Tantalus respected local custom and made a sacrifice of zebu.

At the request of the inhabitants of Antsirabe, Chef-Lieu of the Rural Municipality of the same name, the construction of a new church in this village has been prioritised. It will be completed and inaugurated early in the second quarter of the year 2014. According to the annual report of activities in 2013, the renovation of a temple in Antsirabe was carried out to 60% completion. The work was completed and inaugurated as planned for Easter 2014.

Concerning the public meeting done with the population, the presence of each person was reported to ONE. Twenty-one test-pits were abandoned to respect the forestry use of the Bongomirahavavy site. For another 86 test-pits planned on private properties, their owner has received compensation.

Exploration within the exploration license is not complete and closure procedures have yet to be implemented.

A draft contract between Tantalus and the INSTN was created to establish a baseline study in the Ankingameloka Rural Communes, of Ankaramibe, Antsirabe, of Ambaliha, of Anorotsangana and Bemanevika on the exploration project rare earth pyrochlore and its natural derivatives, in Districts Ambanja (Region DIANA) and Analalava (Sofia region). The study must include radiological measures for the establishment of the baseline study in the entire mining permit type R No. 6698 in order to assess changes in radiation levels due to land use changes related to the extraction of rare earths and the implications of the proposed mining development and reclamation activities. Gaia Oy, an environmental firm from Finland, was hired in 2014 by Tantalus to prepare a baseline environmental study in collaboration with local Malagasy companies. Gaia delivered the Term of Reference (ToR), a prerequisite for Environmental Impact Assessment (EIA). SGS has not had access to the ToR documents or any interim reports for this update of the technical report.

5.8.2 Permits

The original mining license type R No. 6698, obtained on April 18, 2003 (November 20, 2008 according the Environmental permit), for Rural Municipalities Ankingameloko, Anorotsangana, Antsirabe, Bemanevika West and Ambaliha (District Ambanja, Diana Region) and the Rural Municipality of Ankaramibe District Analalava (Sofia Region) listed on the cut of R33 Ankaramibe.

SGS has consulted the Environmental Permit (n°52/09/MEF/ONE/DG/PE) delivered to Tantalus by ONE on November 12, 2009. According to the permit, Tantalus had completed an Environmental Impact Assessment (EIE) in order to obtain the permit.

The National Office for the Environment (ONE) had described the restrictive provisions in the environmental permit dated November 12, 2009. Subject to the satisfaction of the Environmental Management Plan of the Project (PGEP: Plan de Gestion Environmental du Projet) with Workbook Environmental Charges annexed to the Environmental Permit, to penalty sanctions under Articles 34 to 37 of the amended Decree No. 99-954. The environmental permit is valid and subject to compliance with the Book of Environmental Charges until an environmental discharge in the event of closure of the Project (cf. Art. 30 (new) of the amended Decree No. 99 -954). If necessary, until the modification of the actual size of the Project whose cases will be specified by regulation (see Art. 14 (new) of Decree No. 99-954).

SGS has consulted a document called “Addendum to Environmental Engagement Plan”. This report made by Tantalus for ONE described in detail the sampling program. According to the report, the initial Environmental Engagement Plan, which follows the Book of Environmental Charges, did not include the excavation work of test-pits. Some modifications had to be made by Tantalus in order to conduct test-pit excavation. For selection of the location of test-pits, avoiding as much as possible to cut down a tree and it must comply with all restrictions and prohibitions of worship, cultural and ritual, and a local have to accept or not the project to build a planned test-pit. When the test-pit is backfilled, because the cuttings were placed in a pile, it is easy to backfill the pit so as to reconstitute the initial stratification of the soil. Due to the quantity, the fill is compacted by trampling of the soil at intervals.

According to the RSE 1, Tantalus had obtained an authorisation for cutting wood in order to develop an area on the site of Ankatafa by the Ambanja Water and Forestry Cantonment dated June 7, 2010.

6 PROJECT RISKS

Mineral exploration and development are high-risk undertakings. There can be no assurance that the exploration of acquired projects, or any other exploration properties that may be acquired in the future, will result in the discovery of potentially economic Mineral Resources, nor that said Resources can be subsequently converted to a substantial Mineral Reserve. Even if a viable Mineral Reserve is identified, there is no guarantee that it can be economically exploited.

The following risks and opportunities were identified that could affect the future economic outcome of the Project. The following does not include external risks that apply to all exploration and development projects (e.g., changes in metal prices, exchange rates, availability of investment capital, change in government regulations, etc.).

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. To the SGS's knowledge, there are no additional risks or uncertainties that could reasonably be expected to affect the reliability or confidence in the exploration information or mineral resource estimate. SGS is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this Technical Assessment, that could materially affect the current Independent Specialists Report.

6.1 Mining Approvals, Tenure, and Permits

Madagascar appears to be in the final stages of reform of the Mining Code that governs the terms and conditions of the conversion of exploration licence PR 6698 to an exploitation licence. Whilst a valid application for this exploitation licence conversion has been lodged, and the conversion is anticipated following the execution of due process by the Cadastral Mining Office of Madagascar, the delivery timeline and finals terms and conditions, as likely specified by a revised Mining Code, are yet to be confirmed.

6.2 Surface Rights Risks

Harena Resources is required to continue engaging with the local communities (many small communes) and local landowners to gain access to exploration targets and for the development of project infrastructure. The Company is obligated by law to get permission from landowners prior to doing any exploration work. Previous operators have engaged in discussions with local and national governments to establish the framework for continued work on these lands and Herena Resources is advised to continue with this engagement process.

6.3 Exploration Risks

Being an advanced-stage exploration project, with a substantial amount of exploration drilling, test pitting, and a well-established deposit model of mineralisation, the exploration risks associated with the Ampasindava Project are relatively low by exploration standards. In SGS's opinion, the remaining generic and common exploration risks that persist on the Project do not pose a significantly higher risk than any other advanced-stage exploration projects in Madagascar.

6.4 Metallurgical Processing Risks

Recent and current metallurgical test work conducted by Outotec, the University of Toronto and SGS Lakefield has emphasised the amenability of the regolith mineralisation to direct leaching. To mitigate the risk that these samples used in these studies are not representative of the Property as a whole; additional samples that are selected to be representative of grade, geography and material type need to be tested.

This will ensure the reproducibility of the results and to also optimise the recovery of REEs. It is recommended to set up a trial heap leaching program (test mining) with inputs form all stakeholders to test the optimal methods of extraction that limit environmental impact. This will illustrate the amenability of the material to heap leaching and show the expected recoveries in real- world circumstances.

Variability studies should be undertaken on a large number of samples that are representative in terms of geography, soil type, depth and grade. The variability study could be accomplished on a bench scale with material that is already collected and stored in Ambanja; this will help ensure that all the material in the resource is amenable to direct leaching.

It is important to highlight that the eventual processing methodology may not be able to extract all of the commodities of interest recorded in the resource estimate. Further optimization of leaching and development of a downstream processing flowsheet will have a significant impact on the economics of the Project.

6.5 Mineral Resource and Reserve Risks

The current distribution of in-situ Mineral Resources by classification based on tonnages of TREO is approximately 7% Measured, 25% Indicated, and 68% Inferred Mineral Resources at the reported cut-off grade. A significant portion of the contained metal of the Deposit, at the reported cut-off grades for the MRE, is in the Inferred Mineral Resource classification. It is reasonably expected that the majority of Inferred Mineral resources could be upgraded to Indicated Minerals Resources with continued exploration.

The mineralised horizons (mineralised domains) in all zones are relatively well understood. However, due to the limited drilling in some areas, all mineralization zones might be of slightly variable shapes from what have been modeled. A different interpretation from the current mineralization models may adversely affect the current MRE. Continued drilling and the completion of a high-resolution airborne LIDAR (Light Detection and Ranging) survey over the project area for the creation of a high accuracy Digital Terrain Model (DTM) may help define with more precision the shapes of the zones and confirm the geological and grade continuities of the mineralised zones.

6.6 Environmental and Social Risks

The environmental impact to date is largely limited to activities associated with exploration activities. With the availability of existing baseline study information, the Company is in a good position to complete a comprehensive Environmental and Social Impact Assessment (ESIA) Plan. The development of the Project will inevitably impart positive aspects on the local economy in respect of employment and the potential for taxation revenues to be used for further social development, but also runs the risk of causing negative impact on the physical environment recognised for its unique biodiversity.

Previous operators of the Project have obtained all the required environmental permits to conduct exploration activities on the licence and previously employed a full-time environmental scientist to ensure that the physical impact of the activities is kept to a minimum. The Project area itself has had environmental restriction to exploration and mining lifted for all but a very small fraction to the northwest portion of the exploration licence. The preparation of an ESIA and management plan, including the preparation for mine closure and the rehabilitation of the site remain as prior conditions for all mining activities. No mining activities can start (and this will eventually apply also to detailed exploration, i.e. trial mining) without prior approval by the relevant environmental authorities, as per the regulations on environmental protection and the commitments contained in the ESIA.

Harena Resources should continue with the social and environmental programs implemented on the Project that, to date, have included the hiring of local people, community projects and strict environmental procedures (including the rehabilitation of all work sites and the planting of trees and shrubs). This aspect of the Project should be revisited as soon as more immediate requirements are met.

In contrast to previous operators who were contemplating In-situ leaching as a metal extraction method, Harena Resources intends to pursue a pit extraction and heap leach production method. Processing will be via a dedicated heap leach farm which will be environmentally ring-fenced, hence having a smaller environmental impact due to the confinement of all leachate solutions. Progressive rehabilitation of the mining pits will take place, fresh water will be recovered using reverse osmosis and nanofiltration from the leachate waste, and solid precipitate waste will be neutral and recycled or blended. Power will be conventional and will employ renewables wherever possible. Detailed engineering studies and designs for the adaptation of these technologies and methodologies specifically to the Ampasindava Project have not yet been completed. These engineering studies and designs will be incorporated into the Definitive Feasibility Study proposed by Harena Resources. Communication and education related to this selected extraction method will be an important aspect to gaining social licence from the various stakeholder groups involved.

Madagascar has a very diverse and unique biological environment, and there are several groups (local and international) invested in protecting the habitat. Harena Resources should continue discussions with all the NGOs and all levels of government to ensure that there all stakeholders understand the benefits of developing the Project. This includes providing work opportunities for local people and to continue using work methods which limit the impact on the environment for all stages of work. Most of the areas observed were in various stages of regrowth following slash and burn agriculture by the local people. There is an opportunity to actually improve the biodiversity in the area if local people are engaged in the Project and are able purchase their food from more sustainable sources.

6.7 Hydrological Risks

Madagascar is beset with a great amount of rain during the winter (December to March). Field work by is considerably hampered during the rainy season because the clay soil does not absorb water very well and gets very wet and slippery. Relatively small rivers become impassable. As part of the planned Environmental and Social Impact Assessment, Harena Resources will have to look at how to best manage this influx of water. Also, as part of the trial mining and pit excavation proposed above, Harena Resources will need to have a better understanding of how water migrates through the soils.

6.8 Development and Operational Risks

The success of the Ampasindava Project will also depend upon the Company having access to sufficient development capital, being able to maintain title to its projects and obtaining all required approvals for its activities.

Ongoing infrastructure development activities include improvement and maintenance of the existing access roads and bridges as well as construction of new ones. The semi-permanent field camp near Ankatafa and the personnel accommodation and laboratory facility in Ambanja will be developed as required. In addition the Company should continue to prepare infrastructure for the planned trial mining.

The operations may be affected by various other factors, including failure to achieve predicted grades in exploration and mining, operational and technical difficulties encountered in mining; difficulties in commissioning and operating plant and equipment, mechanical failure or plant breakdown, unanticipated metallurgical problems which may affect extraction costs; adverse weather conditions, industrial and environmental accidents, industrial disputes and unexpected shortages or increases in the costs of consumables, spare parts, plant and equipment.

7 PROPOSED DEVELOPMENT ACTIVITIES

The Independent Specialists believe that the Ampasindava Project has sufficient technical merit to justify ongoing exploration and development. Harena Resources has proposed a staged program of advanced economic and environmental studies, metallurgical processing validation test work, and the design and construction of a processing demonstration plant for the Projects over the three years following its acquisition by Citius Resources PLC.

Harena Resources initial development program will mainly focus on the delivery of a Feasibility Study (“FS”), a new Environmental and Social Impact Assessment (“ESIA”) Plan, validation of metallurgical test work with new samples, and the engineering design, planning, and construction of a demonstration plant. These activities and specifically the FS undertaken to a high degree of accuracy could then be used as a basis for raising finance for the construction of a project.

Harena Resources proposed exploration and development expenditure for the first three years following acquisition as outlined above is detailed in Table 7-1.

Table 7-1 Harena Resource Proposed 3-Year Development Budget

Planned Activity	**FYR 2023-2024**	**FYR 2024-2025**	**FYR 2025-2026**	**Total to Jun 2026**
Metallurgical Consultants	$ 69,500	$ -	$ -	$ 69,500
Environmental Consultants	$ 25,000	$ -	$ -	$ 25,000
Feasibility Study	$ 477,500	$ 275,000	$ 70,000	$ 822,500
Government Aide	$ 23,000	$ 15,000	$ 20,000	$ 58,000
Site Staff	$ 27,000	$ 42,000	$ 42,000	$ 111,000
Casual wages	$ 16,500	$ 18,000	$ 18,000	$ 52,500
Training	$ 20,000	$ 30,000	$ 30,000	$ 80,000
Transport	$ 12,000	$ 18,000	$ 18,000	$ 48,000
Exploration	$ 49,000	$ 42,000	$ 42,000	$ 133,000
Demo Plant Start Up	$ 1,269,500	$ 65,000	$ -	$ 1,334,500
Social Program	$ 48,500	$ 40,000	$ 30,000	$ 118,500
ESG Infrastructure	$ 12,000	$ 18,000	$ 18,000	$ 48,000
Capital infrastructure / roads etc.	$ 20,000	$ 24,000	$ 6,000	$ 50,000
Other	$ 20,000	$ 24,000	$ 24,000	$ 68,000
Total	**$ 2,089,500**	**$ 611,000**	**$ 318,000**	**$ 3,018,500**

SGS considers that the exploration and development program and budget proposed by the Company (Table 7-1) is appropriate given the development stage of the Project, having regard to the strategy and priorities of the Company and are based on sound technical merit.

8 CONCLUSIONS

SGS concludes that the Harena Resources Ampasindava Project presents exposure to an attractive development-stage opportunity.

The Ampasindava Ionic clay project displays a pervasive and well-balanced rare earth distribution that is prevalent throughout the deposit. The deposit includes appreciable amounts of the critical rare earths dysprosium, neodymium, praseodymium, and terbium as defined by the U.S. Department of Energy.

The Ampasindava Project is one of relatively few projects of ionic clay hosted REE deposits outside of China. This sets it apart from most of the developers vying to be a stable source of these metals (particularly the HREE) for the western world.

Further exploration and development work is warranted on the Project.

The proposed budget allocations are considered consistent with the development potential of the Project and are considered adequate to cover the costs of the proposed programs. The budgeted expenditures are also considered sufficient to meet the minimum statutory expenditure on the tenure.

The Independent Specialist's Report has been prepared on information available up to August 30th, 2024, and SGS is not aware of any material change to the Company's mineral interests since that date.

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10 DATE AND SIGNATURE PAGE

This report titled “Independent Specialists Report on the Ampasindava Rare Earths Project, Antsiranana Province, Madagascar” dated the August 30th, 2024 (the “Independent Specialists Report”) for Harena Resources Pty Ltd was prepared and signed by the following authors:

The Technical Assessment Date of this Report is August 30th, 2024.
The Effective Date of the current MRE, as detailed in this Report, is November 1st, 2023.

Signed by:

"Original Signed and Sealed"

Independent Specialist / Competent Person	Company
Yann Camus, P.Eng.	SGS Geological Services (“SGS”)

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August 30th, 2024

APPENDIX A - JORC Code, 2012 Edition – Table 1

JORC Code, 2012 Edition – Table 1
Section 1 Sampling Techniques and Data
(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. 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 (e.g. '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 (e.g. submarine nodules) may warrant disclosure of detailed information.*	• *A total of 284 outcrop samples were collected and analysed from within the project area. These were predominantly peralkaline intrusive rocks collected from areas associated with radiometric anomalies. The results returned grades up to 2.24% TREO in peralkaline granites and up to 0.82% TREO in volcanic breccias.* • *A small soil sampling program was undertaken over areas of the property which had no outcrop. The sampling program was superseded by the subsequent trenching program.* • *A total of five trenches have been excavated across the property.* ○ *A single 30 m long trench was excavated across the Ampasibitika prospect in 2008. The trench averaged 0.7 m in depth and didn't reach bedrock. A total of 16 contiguous approximately 2 m long samples were collected from the trench. The highest-grade samples returned average grades of 0.2% TREO.* ○ *Two 100 m long trenches (TANT2 and TANT3) were excavated in the Befitina prospect. The trenches were dug perpendicular to each other and averaged 3 m in depth. A total of 20 contiguous samples approximately 5 m long were collected from TANT2 and 22 contiguous samples, also approximately 5 m long were collected from TANT3. Results from TANT2 were between 0.09% and 0.4% TREO, while results from TANT3 ranged from 0.07% and 0.3% TREO.* ○ *Two 100 m long trenches (TANT4 and TANT5) were excavated from the Caldera prospect. The trenches were dug perpendicular to each other and achieved depths of over 4 m. TANT4 intersected predominately regolith, while TANT5 was entirely within saprolite. A total of 100 contiguous samples approximately 1 m long were collected from each trench. Assay results from TANT4 ranged from 0.05% to 0.3% TREO, while results from TANT5 ranged from 0.05% to 0.5% TREO.* • *A total of 4,474 test pits were excavated across the property between 2011 and 2013. The pits were vertical in orientation and typically 1 m by 1 m. The pits had a vertical depth up to 10 m, with an average depth of 5.68 m. The pits were excavated to bedrock or the final depth of 10 m, whichever was first. Pit spacing was between 50 m and 250 m.* ○ *A total of 13,926 samples were taken in the pedolith (soil) horizon, with assays ranging from 0.003% to 1.5% TREO.* ○ *A total of 16,133 samples were taken in the saprolite horizon, with results ranging from 0.003% to 3.9% TREO.*

		○ The sampling methodology involved marking out the samples on the same wall of each pit at 1.0 m intervals (0.5m lengths are found in the 2011 data). Samples were collected from the lowermost interval first to minimise contamination. Collection involved using the pointed end of a rock pick or machete to create a continuous vertical channel with the displaced material collected in a bucket or a polythene sample bag with an average sample weight of 1.8 kg. A unique, predefined sample tag was then placed into the bag and the bag closed with a plastic cable tie. Once bagged, the samples were manually carried to the field camp, then transferred to the sample preparation facility in Ambanja. ○ To measure the density accurately, a ledge was carved into the wall of the pit on the same day that its depth was excavated. A tube was carefully hammered into the ledge to fill the tube. The tube was dislodged from the wall and excess material was carefully shaved off with a machete. The regolith was carefully emptied into a sample bag and weighed. This bag was carefully sealed and taped to the assay sample bag from the same depth. Once the sample arrived at the preparation lab the density sample was weighed once more to validate the humid weight before drying. As the density sample was dried in ovens, it was weighed at regular intervals to ensure that the sample was completely dry without dehydration of mineralogical volatiles. • A hammer-style soil sample drill was used to drill 47 holes into the regolith to ascertain if it was a better option than the test pits. A total of 354 m was drilled, with hole lengths ranging from 1.5 m to 11 m. The samples collected were not used due to technical issues with their collection.
Drilling techniques	• Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. 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).	• A diamond drill program was completed over the property between July 2010 and October 2011. The drill program was completed using industry standard practices. ○ A total of 277 holes were drilled on the Ampasibitika prospect, comprising NW (7.62 cm core diameter), NTW (5.61 cm core diameter) and BTW (4.17 cm core diameter) core. ○ A total of 20 holes were drilled on the Caldera prospect using NW and NTW core. ○ All holes were drilled from surface and all holes were vertical in orientation. ○ Drillhole collars were surveyed using a consumer-grade handheld GPS. ○ There is no record of how downhole surveys were conducted.
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.	• There is no record of sample recovery from the diamond drilling.
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.	• Drill core was placed in wooden boxes with depth markers, sealed with a wooden lid and transported to the logging facility. • At the logging facility, the core was logged and photographed by company geologists and marked up for sampling.

	• Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged	• The initial sampling strategy was restricted to intersections that included intrusive rocks, and rocks that were radioactive and / or fluoresced under ultraviolet light. However, this led to an incomplete sampling of the bedrock sections of the drillholes. • Subsequent infill sampling was undertaken to ensure proper sampling of the core. • Geological logging was completed for all holes, and it is representative. The lithology, alteration, and structural characteristics of drill samples were logged following standard procedures and using standardised geological codes. • Logging was both qualitative and quantitative depending on field being logged. • All drill-holes were logged in full
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.	• Bedrock core sample: ○ Drill core within bedrock was split using a hydraulic splitter, with one half being sampled and the other half returned to the sample tray. ○ Density measurements using water immersion were undertaken on the half being sampled. ○ The half core was crushed to minus 2 mm in an electric jaw crusher, with cleaning utilising compressed air and a vacuum cleaner between samples to minimise contamination. ○ Crushed samples were split twice in a riffle splitter to produce a 250 g to 350 g homogenised sample, which was bagged for analysis. • Core samples within regolith: ○ Drill core samples within regolith were split with a geological hammer. ○ The samples were weighed, inclusive of moisture, then placed into steel bowls for drying. ○ Once dry, the samples were re-weighed and the weight recorded. ○ If rock fragments were present in the samples, they were crushed to minus 2 mm in an electric jaw crusher. After each sample, blank material was crushed and the equipment cleaned with compressed air and a vacuum cleaner in order to minimise contamination. ○ If the dried samples contained no rock fragments, they were manually pulverised in the stainless steel bowls using a large wooden pestle. ○ Crushed samples were split twice in a riffle splitter to produce a 250 g to 350 g homogenised sample, which was bagged for analysis. • The remaining coarse reject material is retained and stored at the sample preparation facility. The drill core is stored in a dedicated warehouse in Ambanja. • It is the opinion of the Competent Person that the sampling techniques and sample preparation are appropriate for the material being sampled.
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.	• Samples in 2011 were analysed using a 38-element element Li-fusion Induced Coupled Plasma Mass Spectrometry (ICP-MS) analysis at ALS Chemex in Vancouver (ALS code ME-MS81). The analysis comprised both trace element and REE analysis. • Samples in 2013 were analysed at the SGS lab in Boovens, South Africa using a sodium peroxide fusion with an ICP-MS finish. The technique (SGS code GO IMS91B) analyses for 17 REE elements. Because the fusion temperature is lower than that of lithium metaborate fusions, the hydride elements are not volatised.

	- *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	- *Both the ALS Chemex laboratory and the SGS laboratory are ISO accredited. The analytical methods used are according to industry standards and data provided is appropriate for use in the resource estimation.* - *The company implemented their own Quality Assurance and Quality Control (QAQC) procedure, to validate the sample results, whereby one blank, one standard and one duplicate material are inserted within every 35 samples. This relates to an insertion rate of approximately 8 %, with one in every 12 samples sent to the ALS Chemex laboratories in 2011 and SGS South Africa laboratory in 2013.* - *The company developed three standards using material obtained on site. None of the standards were validated by round robin or statistical validation. SGS analysed the results from the 326 standards submitted and determined a failure rate of between 0.3% and 2.5%, depending on the element being analysed. SGS do not consider the failure rate to be material to the MRE.* - *The results for the blanks were similar to those for the standards, but no discernible issues were identified.* - *A total of 569 coarse duplicates were submitted for analysis. The results showed a good level of correlation, although there were some results outside the confidence intervals.* - *It is the opinion of the Competent Person that the QAQC program and results were of sufficient quality to support the MRE.*
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.*	- *The database was continually validated by the company on receipt of assays from the lab. Drilling, pitting, trenching and window sampling collar locations, surveys and logging was entered manually into the database by the Geologist responsible for the specific hole/pit. The data were then validated by a dedicated Database Manager.*
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.*	- *Drillhole collars were surveyed using a consumer-grade handheld GPS. Comparison of the collar data with what was considered the most accurate topographical data demonstrated a discrepancy in the elevation coordinates of the collars. The x- and y- coordinates of the collars were maintained and the z- coordinates were pressed to the topography.* - *SGS were provided with 5 topographical surfaces at various contour intervals and covering different parts of the property. The selected topography was a digitised map at 10 m contours based on a combination of government maps and a Fugro geophysical survey. This survey covered the entire property area and was considered the most accurate.*
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*	- *The spacing of the test pits, at between 50 m and 250 m across the property is considered sufficient spacing for the type and style of mineralisation.* - *The diamond drilling was confined primarily to the Ampasibitika prospect and covered an area of approximately 5 km by 4 km. Drill hole fences were spaced at either 100 m or 200 m and drillholes along the fence were spaced at 50 m intervals. This too is considered sufficient to establish geological and grade continuity.* - *The pedolith and saprolite zones were full-length composited for the MRE.*

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.*	• *The sampling in the test pits and diamond drilling of the pedolith and saprolite is considered in keeping with the broad orientation of the mineralisation, although the test pits and drilling demonstrated that the orientation of the mineralisation was more complex than previously considered.*
Sample security	• *The measures taken to ensure sample security.*	• *At all times samples were in the custody and control of the Company's representatives until delivery to the laboratory where samples were held in a secure enclosure pending processing.*
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data*	• *The Competent Person for Exploration Results reported here has reviewed the field procedures used for sampling program at field and has compiled results from the original sampling and laboratory data.* • *No external audits were undertaken on the data.*

Section 2 Reporting of Exploration Results
(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code 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 comprises one exploration licence (permit PR 6698) made up of 608 contiguous 625 m by 625 m unit blocks that encompass a total area of 237.5 km^2.* • *The permit is currently granted as a "Permis de Recherche" (research permit), or PR, which grants the exclusive right for prospecting and research.* • *An application was submitted to the Malagasy Mining Registry Office ("BCMM") effective September 18, 2020, to convert the PR into a "Permis d'Exploitation" (PE) and the permit is accordingly covered by the protections offered by Article 36 of the Mining Code.*
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• *Between the 1920s and the 1970s, academic research was undertaken across the property and it was mapped by government geologists.* • *Between 1988 and 1991 a Russian-funded exploration programme termed the Soviet Geological Mission was completed in conjunction with the Malagasy Office Militaire National pour les Industries Stratégiques (OMNIS).* • *In 2008 Fugro Consult GmbH (Fugro) was commissioned to undertake field mapping, sediment sampling, outcrop sampling and an airborne magnetic and radiometric survey.*
Geology	• *Deposit type, geological setting and style of mineralisation.*	• *The deposit is envisaged to represent a regolith-hosted REE deposit, where the REEs have been ionically absorbed onto clay minerals in the regolith.* • *The REE-enriched source rocks of the Ambohimirahavavy igneous complex are alkaline and peralkaline granitic dykes and sills, but also includes the more fractionated parts of the complex. Subordinate structures may have acted as preferential pathways for post-intrusive hydrothermal fluids.*

		• *The mineralised source rocks were subject to intense weathering due to the sub-tropical climate (average temperatures of higher than 25°C and rainfall exceeding 2000 mm per year) that resulted in the development of widespread and typically thick regolith.*
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.*	• *All drill hole summary location data is provided in Appendix 1 to this report and is accurately represented in appropriate location maps and drill sections where required.*
Data aggregation methods	• *In reporting Exploration Results, weighting Averaging techniques, maximum and/or minimum grade truncations (e.g. 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.*	• *Sample length weighted averaging techniques have been applied to the sample assay results.* • *No grade top cuts have been applied.*
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 (e.g. 'down hole length, true width not known').*	• *The exploration target is the regolith horizon, which predominantly follows the landform. All drilling and test pits were vertical in nature, which would be considered perpendicular to the mineralisation.*
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.*	• *The 2014 Technical Report contains various maps and figures showing the sample results in the geological context.*
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.*	• *All analytical results for REEs have been 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*	• *All information that is considered material has been reported, including test pit sampling results, drilling results, geological context, etc.*

	samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	- *Metallurgical test work was undertaken by the Soviet Geological Mission between 1988 and 1991.* - *Mineralogical and metallurgical test work was carried out between 2010 and 2012 in Germany, France and Canada.* - *SGS Lakefield (Canada) completed metallurgical testing on 60 samples totalling 982 kg of material in 2013.* - *Outotec conducted leaching experiments on mineralised material in 2014.* - *An airborne magnetic and radiometric survey was flown over part of the property in 2008*
Further work	- *The nature and scale of planned further work (e.g. 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.*	- *The Company plans to undertake further exploration work on the property to validate the exploration to date and further enhance the MRE.*

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 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.*	- *SGS received the test pit and drillhole database from Tantalus on the 21st September 2014 as two Microsoft Excel files, with 90 additional assays received on the 9th September 2014 and a final trench update on the 11 September 2014.* - *The data was manually validated by looking through the collar, survey, assay and lithology files for obvious errors, such as missing data or negative values.* - *The data was imported in SGS's proprietary modelling software, Genesis, which checks for overlaps, data duplication, correct EOH depth, missing assays, incorrect hole names and survey inconsistencies.* - *The trench data and auger hole data were removed due to inconsistencies, together with some test pits and drillholes that were missing coordinate data. All grab sample data was removed from the database prior to modelling.* - *The final DB contained 4771 collars (4412 test pits and 359 drill holes), 37,212 assays and 37,212 lithologies. All drill holes are vertical except for 268 (dip at -45°) and 251 (dip at -70°).* - *There are 19 assays elements available for all of the assay intervals.*
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.*	- *Site visits of the Property were completed by Claude Bisaillon, Eng, SGS Geological Engineer, between 26 April and May 3, 2013, and September 12 and September 28, 2013. A third site visit was conducted by Guy Desharnais Ph.D., P.Geo. between November 28 and December 8, 2013.* - *As no further work has been completed on the property since the 2013 site visits, an updated site visit wasn't considered necessary.*
Geological	- *Confidence in (or conversely, the uncertainty of) the*	- *SGS considers the geological interpretation to be robust.*

interpretation	*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 confidence in the geological interpretation is reflected by the assigned Mineral Resource classification.* - *The geology has guided the resource estimation, particularly the lithological control.* - *Grade and geological continuity are conceptual and will be confirmed with infilled drilling.*
Dimensions	- *The extent and variability of the Mineral Resource expressed as length (along strike otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	- *The mineralisation is approximately 19.9 km long (NW-SE) by 8.4 km wide (NE-SW).* - *The regolith thickness ranges from 0 m to over 40 m, averaging 13.5 m.*
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 (e.g. 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..*	- *The geological and mineralisation interpretation of the deposit as well as the geostatistics, block modelling and resource estimation were made using Genesis, SGS' proprietary modelling software.* - *The following steps were followed for the resource estimation:* ○ *Validation of the drill hole database* ○ *Selection of the mineralised intervals for each drill hole for each layer (PED and SAP)* ○ *Selection of the topography surface to be used for the model* ○ *Creation of volume models of the layers in 2D (2D XY model including thickness and Z position)* ○ *Variogram modeling in 2D for 19 variables for each layer* ○ *Conversion in 3D block model to use in Genesis software (3D XYZ model with PED and SAP tags)* ○ *Estimation of the grades for 19 variables for each layer in the 3D block model* ○ *Classification of the resource according to drill hole spacing* ○ *Creation of solids within barren areas to constrain the resource* ○ *Validation of the density for each layer and for each prospect (Ambaliha, Ampasibitika, Ampasibitika South, Befitina, Caldera and North West Territories)* ○ *Queries on the 3D block model but with cut-off grade applied on the average grade over the total thickness* - *The grid chosen for the 3D block model was 30 m x 30 m x 1 m.*
Moisture	- *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	- *The tonnages were estimated on a dry basis.*
Cut-off parameters	- *The basis of the adopted cut-off grade(s) or quality parameters applied.*	- *The base case cut-off grade (COG) was determined using the list of prices to be used for the evaluation of the reasonable prospect for economic extraction. Then, the processing recoveries and the marginal costs were estimated. A discount of 35% was also applied to the in-situ value to account for last processing stage (REO separation).*
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*	- *No mining assumptions were made regarding the deposit*

	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.	
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.*	- *Metallurgical test work was undertaken on the deposit, but recovery data was inconclusive. For the purposes of the MRE, assumed recoveries were applied to determine the cut-off grade.*
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.*	- *There are no studies available on the environmental impacts of the mining and processing operation.* - *SGS is not aware of any studies being started on the Project*
Bulk density	- *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.*	- *Wet density and dry density measurements were completed across a large proportion of the sample database.* - *There were 21,113 wet density samples, averaging 1.53 g/cm^3 and 4,309 dry density samples averaging 1.12 g/cm^3.* - *There was no discernible difference in the bulk density of the pedolith and saprolite horizons.*
Classification	- *The basis for the classification of the Mineral Resources into varying confidence categories.* - *Whether appropriate account has been taken of all relevant factors (i.e. 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*	- *The classification was done by drawing outlines of areas that have been drilled to a hole/pit grid spacing.* - *A grid spacing of at least 50 m x 50 m corresponds to measured resources.* - *A grid spacing of at least 200 m x 200 m corresponds to indicated resources.* - *A grid spacing of at least 500 m x 500 m corresponds to inferred resources.*

	reflects the Competent Person's view of the deposit.	
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• *A peer review of the block modelling parameters and resource estimation methods has been done by fellow colleagues and competent persons.*
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.*	• *Based on available drilling data, block model validation has shown that the block model fairly reflects the underlying data inputs.* • *The MRE reported is a global estimate with reasonable prospects of eventual economic extraction.* • *There has been no production on the property.*