



**NI 43-101 Technical Report**

**Resources for the Tantalus**

**Rare Earth Ionic Clay Project**

**Northern Madagascar**

Respectfully submitted to:  
Tantalus Rare Earths AG

By: SGS Canada Inc.  
Guy Desharnais, Ph.D., P. Geo  
Yann Camus, Eng.  
Claude Bisailon, Eng  
SGS Canada – Geostat

Effective Date:  
October 20<sup>th</sup>, 2014

**Disclaimer:**

This document is issued by the SGS Canada Inc. under its General Conditions of Service accessible at [http://www.sgs.com/terms\\_and\\_conditions.htm](http://www.sgs.com/terms_and_conditions.htm). Attention is drawn to the limitation of liability, indemnification and jurisdiction issues defined there in. Any holder of this document is advised that information contained herein reflects the Company's findings at the time of its intervention only and within the limits of the Client's instructions, if any. The Company's sole responsibility is to its Client and this document does not exonerate parties to a transaction from exercising their rights and obligations under the transaction documents. Any unauthorized alteration, forgery or falsification of the content or appearance of this document is unlawful and offenders may be prosecuted to the fullest extent of the law

## DATE AND SIGNATURE PAGE

The effective date of this report “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” of Tantalus Rare Earths AG is October 20th, 2014.

***Prepared by:***

(Original copy signed and sealed)

---

***Guy Desharnais, Ph.D., P.Ge.***

***Date November 27th, 2014***

(Original copy signed and sealed)

---

***Yann Camus, Eng.***

***Date November 27th, 2014***

(Original copy signed and sealed)

---

***Claude Bisailon, Eng.***

***Date November 27th, 2014***

## 1 Executive Summary

Tantalus Rare Earths AG ('Tantalus' or 'the Company') holds the exploration licences grant PR 6698 in the province of Antsiranana, Madagascar. Tantalus has mandated SGS Canada Inc. – Geostat Office (hereafter "SGS Geostat") to complete a resource estimate technical report for the PR 6698 licenses ("The Project").

This report presents a technical review of the geology and the mineralization on the property. It includes a summary of previous work, a detailed description of the new geological work carried out by Tantalus, updated resource estimation, and a series of recommendations to advance this project.

Site visits of the Property were completed by Claude Bisailon, 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.

License grant PR 6698 is situated approximately 40 km south-west of Ambanja and covers an area of 300 km<sup>2</sup> in the Antsiranana province. The project is currently held 100% by Tantalum Rare Earth Malagasy SARL in Madagascar, a wholly owned subsidiary of Tantalus. Access to the Property is by road from Ambanja or by boat from Nosy Be.

The TRE property was first recognized for hosting mineralized dykes hosting high grade Rare Earth Elements (REE) in fresh rock. REE hosted within the regolith (or soil profile) was recognized 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 Tantalus 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 TRE 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 4474 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 4412 pits were retained for the resource estimation along with 359 drill holes.

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

**Table 1-1: Resource Base Case – COGs of 300 and 500 ppm TREO excluding Ce (TREOnoCe)**

Classification	Tonnage (t)	Volume (m <sup>3</sup> )	Area (m <sup>2</sup> )	Density (t/m <sup>3</sup> )	Thickness (m)			TREO (ppm)	TREOnoCe (ppm)	CREO (ppm)	HREO (ppm)	LREO (ppm)	HREO / TREOnoCe ratio	Contained TREO * (t)
					Total	PED	SAP							
Measured	40,103,550	35,948,700	6,618,600	1.12	5.4	2.8	2.6	975	660	296	187	788	28%	39,092
Indicated	157,580,640	143,150,400	20,998,800	1.10	6.8	2.6	4.2	878	554	255	166	712	30%	138,292
<b>Measured + Indicated</b>	<b>197,684,190</b>	<b>179,099,100</b>	<b>27,617,400</b>	<b>1.10</b>	<b>6.5</b>	<b>2.7</b>	<b>3.8</b>	<b>897</b>	<b>575</b>	<b>263</b>	<b>170</b>	<b>727</b>	<b>30%</b>	<b>177,383</b>
Inferred	429,999,525	390,900,600	70,396,200	1.10	5.6	2.7	2.9	894	574	247	149	745	26%	384,552

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price

- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

\* Contained TREO is presented as in-situ. Values do not account for recovery losses.

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

CREO = Nd<sub>2</sub>O<sub>3</sub>+Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

In the 2011 Critical Materials Report from the U.S. Department of Energy, it was revealed that five of the 16 rare earth elements were found to be critical in terms of supply risk in the short term (present to 2015) and into the medium term (2015 – 2025). These elements are termed “critical” because of their scarcity. As shown in the criticality matrix below, those elements deemed critical are: Dysprosium, Europium, Neodymium, Terbium, and Yttrium. Since publication of this report, we have seen a significant drop in the prices of all REE; however the Critical REE (CREE) have retained their value better than their “non-critical” counterparts.

The Tantalus Ionic clay project includes appreciable amounts of critical rare earths Dysprosium, Europium, Neodymium, Terbium, and Yttrium. 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.

Table 1-2 shows the tonnage of the individual oxides contained in the project mineral resources along with the distribution of the tonnages in Figure 1-1. Taking into account the individual REO recoveries and values (detailed in section 14), the distribution of the potential revenues generated by each REO is shown in Figure 1-2.

**Table 1-2: Tonnage of the Individual Oxides Contained in the Project Mineral Resources**

Classification	Y <sub>2</sub> O <sub>3</sub> (t)	La <sub>2</sub> O <sub>3</sub> (t)	Ce <sub>2</sub> O <sub>3</sub> (t)	Pr <sub>2</sub> O <sub>3</sub> (t)	Nd <sub>2</sub> O <sub>3</sub> (t)	Sm <sub>2</sub> O <sub>3</sub> (t)	Eu <sub>2</sub> O <sub>3</sub> (t)	Gd <sub>2</sub> O <sub>3</sub> (t)	Tb <sub>2</sub> O <sub>3</sub> (t)	Dy <sub>2</sub> O <sub>3</sub> (t)	Ho <sub>2</sub> O <sub>3</sub> (t)	Er <sub>2</sub> O <sub>3</sub> (t)	Tm <sub>2</sub> O <sub>3</sub> (t)	Yb <sub>2</sub> O <sub>3</sub> (t)	Lu <sub>2</sub> O <sub>3</sub> (t)	TREO (t)
Measured	4,520	9,667	12,620	1,876	6,347	1,084	115	923	133	766	144	415	58	369	55	39,092
Indicated	15,953	30,677	51,031	6,110	20,659	3,661	423	2,837	440	2,630	522	1,507	222	1,409	210	138,292
<b>Measured + Indicated</b>	<b>20,472</b>	<b>40,344</b>	<b>63,651</b>	<b>7,986</b>	<b>27,006</b>	<b>4,745</b>	<b>538</b>	<b>3,760</b>	<b>573</b>	<b>3,397</b>	<b>666</b>	<b>1,922</b>	<b>280</b>	<b>1,779</b>	<b>265</b>	<b>177,383</b>
Inferred	38,745	95,894	137,928	17,960	59,110	9,468	1,038	7,578	1,097	6,384	1,235	3,645	521	3,431	517	384,552

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

Contained TREO is presented as in-situ. Values do not account for recovery losses.

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

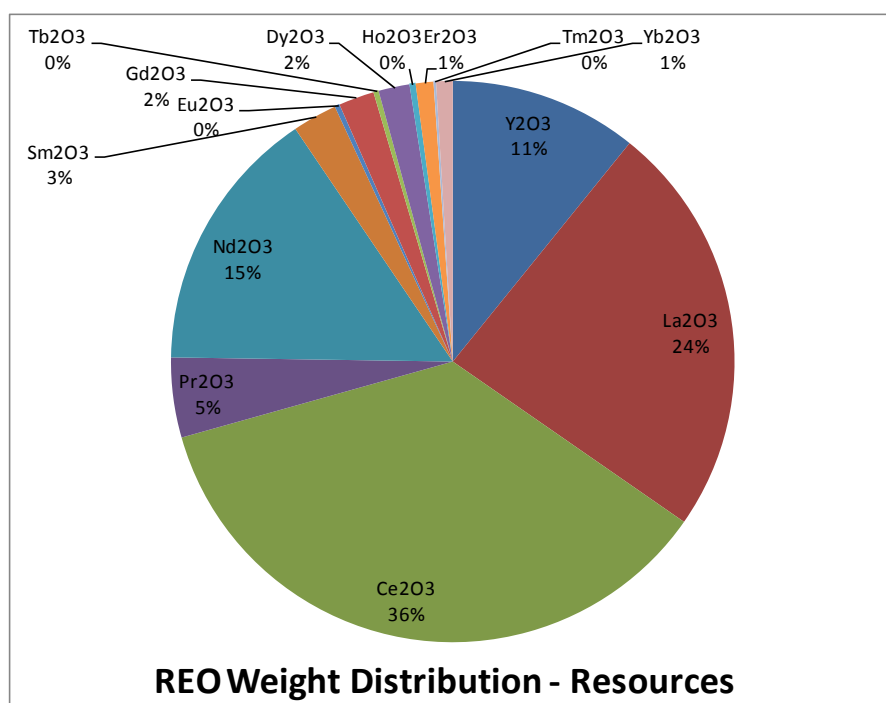


Figure 1-1: Distribution of the Individual Oxides Contained in the Project Mineral Resources

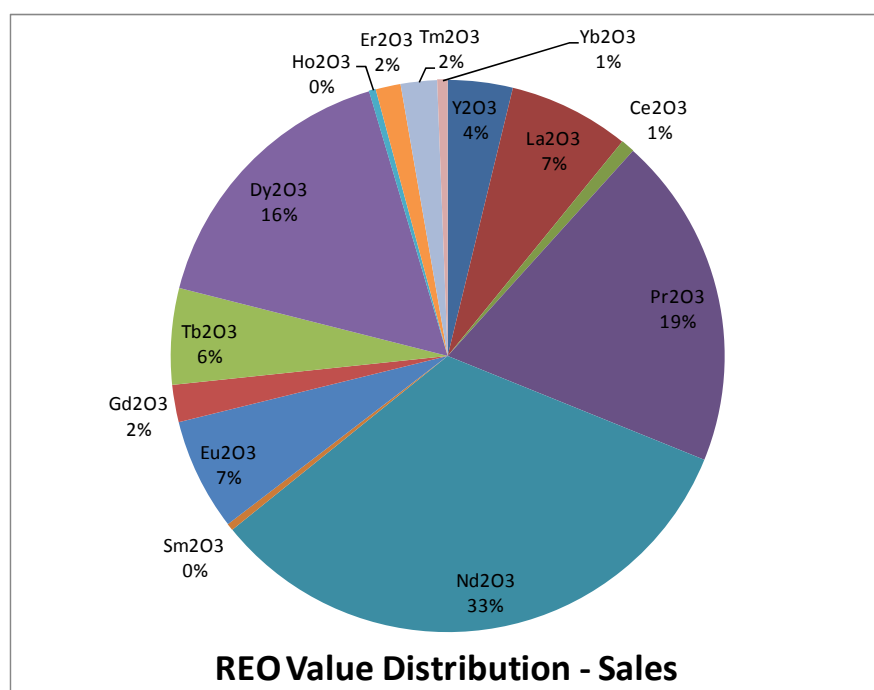


Figure 1-2: Distribution of the potential revenues generated by the individual REO. Please see section 14 for assumptions used to establish these values.

This report represents the first update to the mineral resource first published by SRK in 2013. The resource base has increased significantly since the maiden estimation, in large part due to the test pitting completed in the North West Territory. This report also provides the first resources in the measured and indicated resources and the first application of a cut-off grade to resources in the ionic clays for this project. The TRE property is significant because it hosts a vast resource of ionic clays whose value is driven by the critical group of rare earth elements.

Work to date on the TRE project has focused on outlining resources; this goal has largely been completed with a significant mineral inventory that can be incorporated in economic studies. Recommendations are presented in section 20 of this report. Many studies should be conducted in parallel at this stage to mitigate the various risks known for this project as outlined in sections 19 and 20 of this report. These studies should culminate in a Feasibility Study which will provide further detail on the economic potential for this project.

## TABLE OF CONTENTS

<b>1</b>	<b><i>Executive Summary</i></b> .....	<b><i>iv</i></b>
<b>2</b>	<b><i>Introduction</i></b> .....	<b>1</b>
2.1	General .....	1
2.2	Terms of Reference.....	2
2.3	Currency, Units, Abbreviations and Definitions.....	2
2.4	NI 43-101 Disclosure .....	3
<b>3</b>	<b><i>Reliance on Other Experts</i></b> .....	<b>3</b>
<b>4</b>	<b><i>Property Description and Location</i></b> .....	<b>4</b>
4.1	Location .....	4
4.2	Ownership.....	5
4.3	Royalties .....	5
4.4	Permits .....	5
4.5	Environmental Liabilities.....	7
<b>5</b>	<b><i>Accessibility, Climate, Local Resources, Infrastructure and Physiography</i></b> .	<b>8</b>
5.1	Physiography.....	8
5.2	Accessibility .....	10
5.3	Climate .....	10
5.4	Local Resources and Infrastructure.....	10
5.5	Surface Rights.....	11
<b>6</b>	<b><i>History</i></b> .....	<b>12</b>
6.1	Prior Ownership of the Property and Ownership Changes.....	12
6.2	Mineral Exploration Work.....	12
6.2.1	Colonial Exploration.....	12
6.2.2	Soviet Exploration.....	12
6.2.3	Contemporary Exploration and Results .....	13
<b>7</b>	<b><i>Geological Setting and Mineralization</i></b> .....	<b>17</b>
7.1	Regional Geology.....	17
7.2	Property Geology .....	17
7.2.1	Lithology .....	17
7.2.2	Structures.....	23
7.2.3	Regolith .....	24

<b>7.3</b>	<b>Rare Earth Elements and Rare Metals</b> .....	<b>27</b>
<b>7.4</b>	<b>Mineralization</b> .....	<b>28</b>
7.4.1	Introduction.....	28
7.4.2	Bedrock Hosted REE Mineralisation.....	28
7.4.3	Regolith Hosted REE Mineralisation.....	31
<b>7.5</b>	<b>Mineralisation Model</b> .....	<b>33</b>
<b>7.6</b>	<b>Exploration Characteristics</b> .....	<b>33</b>
<b>8</b>	<b><i>Deposit Types</i></b> .....	<b>34</b>
<b>8.1</b>	<b>Deposit Summary</b> .....	<b>34</b>
<b>8.2</b>	<b>Geological Characteristics</b> .....	<b>34</b>
<b>8.3</b>	<b>Mineralisation Characteristics</b> .....	<b>34</b>
<b>8.4</b>	<b>Critical Rare Earth Elements</b> .....	<b>35</b>
<b>9</b>	<b><i>Exploration</i></b> .....	<b>36</b>
<b>9.1</b>	<b>Outcrop Sampling</b> .....	<b>36</b>
<b>9.2</b>	<b>Soil Sampling</b> .....	<b>37</b>
<b>9.3</b>	<b>Trenching</b> .....	<b>38</b>
<b>9.4</b>	<b>Pitting</b> .....	<b>40</b>
9.4.1	Pit Logging and Sampling Procedures.....	41
<b>9.5</b>	<b>Window Sampling</b> .....	<b>42</b>
<b>9.6</b>	<b>Exploration conclusions</b> .....	<b>43</b>
<b>10</b>	<b><i>Drilling</i></b> .....	<b>44</b>
<b>10.1</b>	<b>Core Drilling</b> .....	<b>44</b>
<b>10.2</b>	<b>Core Logging and Sampling Procedures</b> .....	<b>45</b>
<b>10.3</b>	<b>Core Drilling Summary</b> .....	<b>45</b>
<b>11</b>	<b><i>Sample Preparation, Analyses and Security</i></b> .....	<b>46</b>
<b>11.1</b>	<b>Sample Quality Assurance and Quality Control (QAQC)</b> .....	<b>46</b>
11.1.1	Standards .....	46
11.1.2	Blanks .....	50
11.1.3	Duplicates.....	54
<b>11.2</b>	<b>Sample Preparation and Analysis</b> .....	<b>59</b>
11.2.1	Drill Core Samples - Bedrock Intersections .....	59
11.2.2	Drill Core Samples - Regolith Intersections .....	60
11.2.3	Pit and Window Samples.....	60
11.2.4	Analytical Method .....	61
11.2.5	Topographical Data .....	61

11.3	<b>Data Quality Summary</b> .....	61
<b>12</b>	<b>Data Verification</b> .....	<b>62</b>
<b>13</b>	<b>Mineral Processing and Metallurgical Testing</b> .....	<b>63</b>
13.1	<b>Historical Testwork</b> .....	<b>63</b>
13.1.1	Soviet Mineralogical Testwork .....	63
13.1.2	Soviet Metallurgical Testwork .....	64
13.2	<b>Pre 2013 Testwork</b> .....	<b>64</b>
13.2.1	Mineralogical Testwork.....	64
13.2.2	University of Toronto Testwork .....	67
13.2.3	Two-Stage Leaching Experiments:.....	73
13.3	<b>Post 2014 Testwork</b> .....	<b>75</b>
13.3.1	Outotec Testwork .....	75
13.3.2	SGS Lakefield Testwork .....	77
13.3.3	SGS Lakefield Hydrometallurgical Testing .....	79
13.3.4	SGS Lakefield Heap Leaching Tests .....	87
13.3.5	SGS Lakefield Liquor Treatment Tests.....	93
13.3.6	SGS Lakefield Conclusions and Recommendations.....	96
13.4	<b>Metallurgical Conclusions</b> .....	<b>97</b>
<b>14</b>	<b>Mineral Resource Estimates</b> .....	<b>99</b>
14.1	<b>Introduction</b> .....	<b>99</b>
14.2	<b>Validation of the Database</b> .....	<b>99</b>
14.3	<b>Selection of the Mineralized Intervals</b> .....	<b>100</b>
14.4	<b>Validation of the Density</b> .....	<b>102</b>
14.5	<b>Selection of the Topography Surface to be Used for the Model</b> .....	<b>102</b>
14.6	<b>Volume Modeling of the Layers in 2D</b> .....	<b>103</b>
14.7	<b>Conversion of the 2D Volume Modeling into a 3D Block Model</b> .....	<b>103</b>
14.8	<b>Estimation of the Grades in the 3D Block Model</b> .....	<b>105</b>
14.8.1	Capping of the Assays.....	105
14.8.2	Modeling of Variograms.....	106
14.8.3	Estimation by Kriging.....	111
14.9	<b>Classification</b> .....	<b>112</b>
14.9.1	Definitions.....	112
14.9.2	Classification Method .....	113
14.10	<b>Inclusion of Permit Outlines to Constrain the Resource</b> .....	<b>114</b>
14.11	<b>Inclusion of “Dead Zones” to Constrain the Resource</b> .....	<b>114</b>
14.12	<b>Delimitation of Sloping Areas vs Flat Areas</b> .....	<b>115</b>

<b>14.13</b>	<b>Separation of the Resource into the Different Prospects .....</b>	<b>116</b>
<b>14.14</b>	<b>Base Case for the Estimation of the Resources .....</b>	<b>116</b>
14.14.1	Determination of the Cut-Off Grade for the Base Case of the Resources .....	116
14.14.2	Resource Tables for the Base Case .....	118
14.14.3	Distribution of the Oxides Contained in the Resources, in the Concentrate, and Potential Value for the Base Case .....	122
<b>14.15</b>	<b>Estimation of the Resources – Sensitivity Analysis .....</b>	<b>124</b>
<b>15</b>	<b><i>Market Studies and Contracts.....</i></b>	<b>127</b>
<b>16</b>	<b><i>Environmental Studies, Permitting and Community Impact .....</i></b>	<b>128</b>
16.1	Summary of Available Reports.....	128
16.2	Permits.....	129
16.3	Environment and Community Conclusions .....	130
<b>17</b>	<b><i>Adjacent Properties.....</i></b>	<b>131</b>
<b>18</b>	<b><i>Other Relevant Data and Information.....</i></b>	<b>132</b>
18.1	Country Profile .....	132
18.1.1	Introduction.....	132
18.2	Mining Industry in Madagascar .....	135
18.2.1	Mining and Exploration Companies .....	136
<b>19</b>	<b><i>Interpretation and Conclusions .....</i></b>	<b>138</b>
19.1	Metallurgical processing.....	138
19.2	Environmental issues .....	138
19.3	Hydrology .....	138
19.4	Economic potential .....	139
19.5	Ionic Clays REE Deposits .....	139
<b>20</b>	<b><i>Recommendations .....</i></b>	<b>140</b>
20.1	Additional Metallurgical Test Work .....	140
20.2	LIDAR survey .....	140
20.3	Infrastructure Development .....	140
20.4	Social and Environmental Programs .....	140
20.5	Proposed Budget for Recommended Work.....	142
<b>21</b>	<b><i>References .....</i></b>	<b>143</b>
	<b><i>Appendix A. Certificate of Qualified Person Guy Desharnais.....</i></b>	<b>146</b>

**Appendix B. Certificate of Qualified Person Yann Camus..... 147**

**Appendix C. Certificate of Qualified Person Claude Bisailon ..... 148**

## LIST OF FIGURES

<i>Figure 1-1: Distribution of the Individual Oxides Contained in the Project Mineral Resources</i> .....	vi
<i>Figure 1-2: Distribution of the potential revenues generated by the individual REO. Please see section 14 for assumptions used to establish these values.</i> .....	vi
<i>Figure 2-1: Map showing the prospects of the Tantalus Project (TREM, 2013)</i> .....	1
<i>Figure 4-1: Map showing the prospects of the Tantalus Project (TREM, 2014)</i> .....	4
<i>Figure 4-2: Map of PR6698</i> .....	6
<i>Figure 5-1: Photograph of the Ambohimirahavavy igneous complex caldera (Gilbertson, 2013)</i> .....	8
<i>Figure 5-2: Photograph of a general view of the eastern part of the Tantalus project taken in December 2013. First image shows a sector burned to prepare for planting rice with a bamboo stand in the bottom of the valley. Second image is taken from the eastern limit of the caldera showing a patchwork of vegetation from the rotating agriculture usage.</i> .....	9
<i>Figure 5-3: Photograph of the Ankatafa camp on the coast.</i> .....	11
<i>Figure 6-1: Map showing the radiometric Ternary imagery for the Tantalus project (after Earthmaps Consulting, 2009)</i> .....	15
<i>Figure 6-2: Map showing the radiometric Ternary imagery for the Tantalus project (after Earthmaps Consulting, 2009)</i> .....	16
<i>Figure 7-1: Map showing the simplified geological setting of north-western Madagascar (after Ganzeev and Grechischev, 2003)</i> .....	18
<i>Figure 7-2: Map showing the geological interpretation for the Tantalus project area (after Earthmaps Consulting, 2003)</i> .....	20
<i>Figure 7-3: Map showing the geological setting for the Tantalus project area (after BGS-USGS, 2008)</i> .....	21
<i>Figure 7-4: Schematic cross-section of the Ambohimirahavavy igneous complex (modified from OMNIS-SM, 1992) (Gilbertson, 2013)</i> .....	22
<i>Figure 7-5: Map showing interpreted structural setting for the Tantalus project area (after Earthmaps Consulting, 2009)</i> .....	24
<i>Figure 7-6: Schematic cross-section of the Tantalus project regolith profile (Gilbertson, 2013)</i> .....	25
<i>Figure 8-1: Short and Medium term Criticality Matrix</i> .....	35
<i>Figure 9-1: Map showing the locations of the Tantalus outcrop and soil samples</i> .....	36
<i>Figure 9-2: Map showing the locations of the Tantalus trenches and drillholes</i> .....	38
<i>Figure 9-3: Photos of work undertaken in the field. 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</i> .....	40
<i>Figure 9-4: Topographic image showing the distribution of pits within the TRE project. 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 colors indicate higher elevation.</i> .....	41
<i>Figure 9-5: Window sampler demonstration. 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.</i> .....	43
<i>Figure 10-1: Photograph of the Versadrill Kmb.4km drill rig in operation (Gilbertson, 2013)</i> .....	44
<i>Figure 11-1: Tantalus Standard – TREO</i> .....	47
<i>Figure 11-2: Tantalus Standard – Ce</i> .....	48
<i>Figure 11-3: Tantalus Standard – Dy</i> .....	48

Figure 11-4: Tantalus Standard – Eu.....	48
Figure 11-5: Tantalus Standard – Nd.....	49
Figure 11-6: Tantalus Standard – Pr.....	49
Figure 11-7: Tantalus Standard – Tm.....	49
Figure 11-8: Tantalus Standard – U.....	50
Figure 11-9: Tantalus Standard – Y.....	50
Figure 11-10: Tantalus Blank – TREE.....	51
Figure 11-11: Tantalus Blank – Ce.....	51
Figure 11-12: Tantalus Blank – Dy.....	51
Figure 11-13: Tantalus Blank – Eu.....	52
Figure 11-14: Tantalus Blank – Nb.....	52
Figure 11-15: Tantalus Blank – Nd.....	52
Figure 11-16: Tantalus Blank – Pr.....	53
Figure 11-17: Tantalus Blank – Th.....	53
Figure 11-18: Tantalus Blank – Tm.....	53
Figure 11-19: Tantalus Blank – Y.....	54
Figure 11-20: Tantalus Blank – Yb.....	54
Figure 11-21: Original v Duplicate TREE (ppm) assays.....	55
Figure 11-22: Original v Duplicate Ce (ppm) assays.....	55
Figure 11-23: Original v Duplicate Dy (ppm) assays.....	55
Figure 11-24: Original v Duplicate Eu (ppm) assays.....	56
Figure 11-25: Original v Duplicate Nb (ppm) assays.....	56
Figure 11-26: Original v Duplicate Nd (ppm) assays.....	56
Figure 11-27: Original v Duplicate Pr (ppm) assays.....	57
Figure 11-28: Original v Duplicate Th (ppm) assays.....	57
Figure 11-29: Original v Duplicate Tm (ppm) assays.....	57
Figure 11-30: Original v Duplicate U (ppm) assays.....	58
Figure 11-31: Original v Duplicate Y (ppm) assays.....	58
Figure 13-1: Optimum Eluant REE Extractions.....	84
Figure 13-2: Isotherms REE Extractions.....	86
Figure 13-3: TREE, Y, Nd and Dy Isotherms.....	87
Figure 13-4: Column Tests Agglomerated Feed - Column 1 (left) and Column 2 (right).....	88
Figure 13-5: Photograph of Column Set-up.....	89
Figure 13-6: Heap Leaching - Main Elements Extractions vs. Time.....	91
Figure 13-7: Heap Leaching Extractions vs. Optimum Eluant Extractions.....	92
Figure 13-8: Heap Leaching - Column 1 (left) and Column 2 (right) Discharge TREE tenors.....	92
Figure 13-9: Aluminium Removal - Precipitation vs. pH.....	96
Figure 14-1: Normal View of Test pits and drill holes with Mineralized Intervals (MIs).....	101
Figure 14-2: View of Test pits and drill holes with Mineralized Intervals shown with a 10x vertical exaggeration.....	101
Figure 14-3: View of Test pits and drill holes with Mineralized Intervals (MIs) with a 10x vertical exaggeration.....	104
Figure 14-4: Perspective View on the BM with Colors in Function of TREOnoCe.....	104
Figure 14-5: Perspective View on the BM with Colors in Function of TREOnoCe (Zoom).....	105
Figure 14-6: Histogram and Cumulative Frequency (Nd Assays – “PED” and “SAP” Layers).....	106
Figure 14-7: Variogram for Nd in the PED Layer.....	107

---

<i>Figure 14-8: Variogram for Pr in the PED Layer .....</i>	<i>108</i>
<i>Figure 14-9: Variogram for Dy in the PED Layer.....</i>	<i>108</i>
<i>Figure 14-10: Variogram for Eu in the PED Layer.....</i>	<i>109</i>
<i>Figure 14-11: Variogram for Nd in the SAP Layer.....</i>	<i>109</i>
<i>Figure 14-12: Variogram for Pr in the SAP Layer.....</i>	<i>110</i>
<i>Figure 14-13: Variogram for La in the SAP Layer .....</i>	<i>110</i>
<i>Figure 14-14: Variogram for Eu in the SAP Layer.....</i>	<i>111</i>
<i>Figure 14-15: Classification of the Resource Using Test pits and drill holes Grid .....</i>	<i>114</i>
<i>Figure 14-16: Dead Zones Outlines Where Test pits and drill holes are Missing .....</i>	<i>115</i>
<i>Figure 14-17: Flat Areas Where the Topography Dips Less Than 5 degrees.....</i>	<i>115</i>
<i>Figure 14-18: Prospects Outlines.....</i>	<i>116</i>
<i>Figure 14-19: Plan view of the blocks included in the base case resource .....</i>	<i>121</i>
<i>Figure 14-20: Distribution of the Individual Oxides Contained in the Project Mineral Resources .....</i>	<i>122</i>
<i>Figure 14-21: Distribution of the Individual Oxides Contained in the Potential Concentrates .....</i>	<i>123</i>
<i>Figure 14-22: Distribution of the Potential Revenues Generated by the Individual Oxides.....</i>	<i>123</i>
<i>Figure 14-23: Tonnage and TREO Grade Variation with Different Cut-Off Grades .....</i>	<i>124</i>

## LIST OF TABLES

Table 1-1: Resource Base Case – COGs of 300 and 500 ppm TREO excluding Ce (TREOnoCe).....	v
Table 1-2: Tonnage of the Individual Oxides Contained in the Project Mineral Resources .....	v
Table 2-1: List of abbreviations .....	2
Table 4-1: Coordinates for the Tantalus project (local Laborde projection) .....	5
Table 6-1: Exploration completed as part of the Soviet Geological Mission (after OMNIS-SM, 1989; 1990; 1992b; 1992c; 1992d; 1992e) .....	13
Table 6-2: Fugro 'Mini bulk sample' results (Fugro, 2008) .....	14
Table 7-1: Rare Earth Elements and their uses. ....	28
Table 9-1: Summary of the Tantalus outcrop sample results (Gilbertson, 2013).....	37
Table 9-2: Summary of the Tantalus trench sample results (Gilbertson, 2013).....	39
Table 9-3: Summary of analyses from the Tantalus pitting campaigns. ....	41
Table 11-1: Summary of the statistical analysis for REE at the Tantalus Project .....	47
Table 11-2: Summary of performance for duplicates .....	59
Table 13-1: Summary of the XRD results for sample TANT2-477067.....	65
Table 13-2: Summary of the XRD results for sample I679066 - I679069 (Gilbertson, 2013).....	66
Table 13-3: Summary of the samples provided to the University of Toronto (Moldoveanu, 2013).....	67
Table 13-4: University of Toronto sample descriptions and aqua regia digestion results (Moldoveanu, 2013) .....	69
Table 13-5: Total Rare Earth Oxide (TREO) results (as wt. %) (Moldoveanu, 2013) .....	69
Table 13-6: Relative Rare Earth Oxide (REO) results (as wt. %) (Moldoveanu, 2013).....	70
Table 13-7: REE extraction levels (as % Extraction) both as individual REE and Total REE, respectively, based on solids analysis (0.5M (NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub> , 60 min, 22°C, S/L = 1/2, pH ~ 5.4) (Moldoveanu, 2013).....	71
Table 13-8: % REE Extraction during leaching with 1M NaCl (Moldoveanu, 2013).....	72
Table 13-9: % REE Extraction during leaching with simulated seawater (0.48M Na) (Moldoveanu, 2013) .....	73
Table 13-10: Two-stage leaching for MC3 (22°C, 60 min, S/L = 1/2) (Moldoveanu, 2013) .....	74
Table 13-11: Sample Inventory.....	77
Table 13-12: Sample Inventory (Cont.).....	78
Table 13-13: Master Clay Composite - Head Assays.....	79
Table 13-14: Contact Methodology - Test Conditions .....	80
Table 13-15: Contact Methodology – REE Extraction (extraction values are reported as %).....	81
Table 13-16: Contact Methodology - Residue Assays .....	82
Table 13-17: Contact Methodology - PLS Assays.....	82
Table 13-18: Optimum Eluant - Test Conditions .....	83
Table 13-19: Optimum Eluant - Extractions (values reported in %).....	83
Table 13-20: Optimum Eluant - Residue Assays.....	83
Table 13-21: Optimum Eluant - PLS Assays.....	84
Table 13-22: Isotherms - Test Conditions .....	85
Table 13-23: Isotherms - Extractions, Residue and Solution Assays .....	86
Table 13-24: Heap Leaching - Main Parameters .....	90
Table 13-25: Column 1 - Extractions (values reported as %).....	90
Table 13-26: Column 2 - Extractions (values reported as %).....	91
Table 13-27: Column 1 - Discharge, Wash and Residue Assays.....	93

<i>Table 13-28: Column 2 - Discharge, Wash and Residue Assays.....</i>	<i>93</i>
<i>Table 13-29: Liquor Treatment - Feed Assays.....</i>	<i>94</i>
<i>Table 13-30: Oxalic Acid Precipitation - Test Conditions .....</i>	<i>94</i>
<i>Table 13-31: Oxalic Acid Precipitation - Precipitation Efficiency (reported as %).....</i>	<i>94</i>
<i>Table 13-32: Aluminium Removal - Test Conditions .....</i>	<i>95</i>
<i>Table 13-33: Aluminium Removal - Precipitation Efficiency (reported as %).....</i>	<i>95</i>
<i>Table 14-1: Variables in the DB and Used in the Resource Estimation.....</i>	<i>100</i>
<i>Table 14-2: Calculated Variables.....</i>	<i>100</i>
<i>Table 14-3: Count and Lengths of Mineralized Intervals, Contained Assays, and Gaps .....</i>	<i>101</i>
<i>Table 14-4: Statistics on Wet and Dry Densities and Water Contents for Each Prospect .....</i>	<i>102</i>
<i>Table 14-5: Densities Used for the Resource Estimation.....</i>	<i>102</i>
<i>Table 14-6: Block Model Size and Coordinates .....</i>	<i>103</i>
<i>Table 14-7: List of Variograms Used for the Estimates .....</i>	<i>107</i>
<i>Table 14-8: Rare Earth Oxides Price List Used for the Cut-Off Grade Determination .....</i>	<i>117</i>
<i>Table 14-9: Rare Earth Oxides Recoveries Used for the Cut-Off Grade Determination.....</i>	<i>117</i>
<i>Table 14-10: Cut-Off Grade Final Determination .....</i>	<i>117</i>
<i>Table 14-11: Resource Base Case – COGs of 300 and 500 ppm TREOnoCe .....</i>	<i>118</i>
<i>Table 14-12: Base Case – Resource Details by Prospect .....</i>	<i>119</i>
<i>Table 14-13: Base Case – Resource Details by Prospect with individual REOs.....</i>	<i>120</i>
<i>Table 14-14: Tonnage of the Individual Oxides Contained in the Project Mineral Resources .....</i>	<i>122</i>
<i>Table 14-15: Tonnage of the Individual Oxides Contained in the Potential Concentrates .....</i>	<i>123</i>
<i>Table 14-16: Cut-Off Grades Used for the Sensitivity Analysis .....</i>	<i>124</i>
<i>Table 14-17: Base Case +100% – Resource Details by Prospect .....</i>	<i>125</i>
<i>Table 14-18: Base Case +100% – Resource Details by Prospect with individual REOs.....</i>	<i>126</i>
<i>Table 20-1: Proposed budget for work to be completed in the next phase of work. ....</i>	<i>142</i>

## 2 Introduction

### 2.1 General

Tantalus Rare Earths AG ('Tantalus' or 'the Company') is a German incorporated holding company (company number HRB 201757), located in Munich. Tantalus is engaged in the exploration and development of rare earth elements.

The company currently holds the Tantalus Rare Earth (TRE) Project, 100% through Tantalum Rare Earth Malagasy SARL (Madagascar), which is a 100% owned subsidiary of Tantalum Holding Ltd (Mauritius), which is a 100% owned subsidiary of Tantalus. The TRE project is located in the Antsiranana Province of north-western Madagascar, approximately 40 km south-west of Ambanja and covers an area of 300 km<sup>2</sup>.

The concession area is comprised of rare earth element (REE)-tantalum-niobium-zirconium-hafnium enriched dykes, sills and argillaceous laterites. The laterites are medium to Heavy REE (HREE) enriched ionic clays; this geological setting represents the primary source of HREE in the world. The project currently includes six principal prospects (Ampasibitika, Ambaliha, Befitina, Caldera, Ampasibitika South and Northwest Territories) that were largely delineated on the basis of historical exploration and airborne geophysical anomalies.

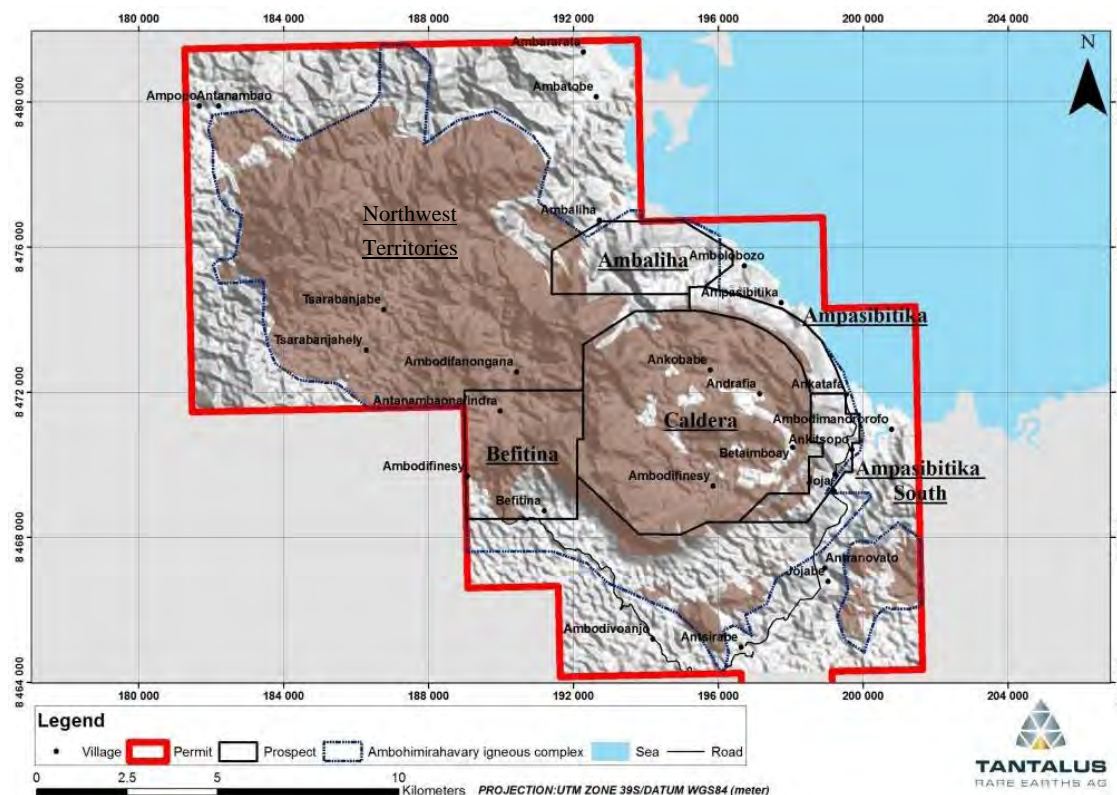


Figure 2-1: Map showing the prospects of the Tantalus Project (TREM, 2013)

Various exploration activities have been undertaken on the Project area with the aim to define the mineral resource in accordance with NI 43-101. The most significant investment to date has been the execution of 4474 test pits that have an average depth of 5.68m and are on a grid spacing of 50 to 250m depending on the zone. A total of 4412 test pits were retained for the resource estimation along with 359 drill holes.

## 2.2 Terms of Reference

SGS Canada Inc ('SGS') has been retained by Tantalus to prepare a technical report and resource estimate of the TRE Project in accordance with the Canadian National Instrument 43-101 (NI 43-101). The information disclosed in this report is derived from previous technical reports, information provided by Tantalus, the internet and three site visits to the Project.

## 2.3 Currency, Units, Abbreviations and Definitions

All measurements in this report are presented in the metric system. Monetary units are in United States dollars (USD\$) unless otherwise specified. The coordinate system is Universal Trans Mercator WGS84.

Table 2-1: List of abbreviations

<b>Abbreviation</b>	<b>Units</b>	<b>Abbreviation</b>	<b>Units</b>
t	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 <sup>3</sup>	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

## 2.4 NI 43-101 Disclosure

The technical information in this report has been prepared in accordance with Canadian regulatory requirements by independent Qualified Persons, or under the supervision of, as set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ('NI 43-101').

The Mineral Resource estimates set out in this report were classified according to the CIM Definition Standards - For Mineral Resources and Mineral Reserves (as adopted by CIM Council in November 2010). Readers are advised that Mineral Resources do not demonstrate economic viability. Mineral Resource estimates do not account for mineability, selectivity, mining loss and dilution. These Mineral Resource estimates include Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that Inferred Mineral Resources will be converted to Measured and Indicated categories through further drilling, or into Mineral Reserves, once economic considerations are applied. Technical information in this report was reviewed and adopted by all Qualified Persons.

## 3 Reliance on Other Experts

This report was prepared in accordance with the NI 43-101, for the TRE Project of Tantalus.

Much of the background information has been sourced from the previous Technical report by SRK (Gilbertson 2013).

SGS has also provided information, drawn conclusions and made estimates to the best of their knowledge, based upon reviewing the following:

- Information provided by Tantalus and their technical staff;
- Public information available at the time of preparation;
- Personal inspections of the Project;
- External data;
- Assumptions, conditions and qualifications established in this report.

Contributions by Dominique Rakotomanana, Mahefa Ratefiarijaona and Montaharison Rakotoandriana for their support in preparing the resource estimation and technical report are duly noted and appreciated.

## 4 Property Description and Location

### 4.1 Location

The Tantalus 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 (TREM, 2014).

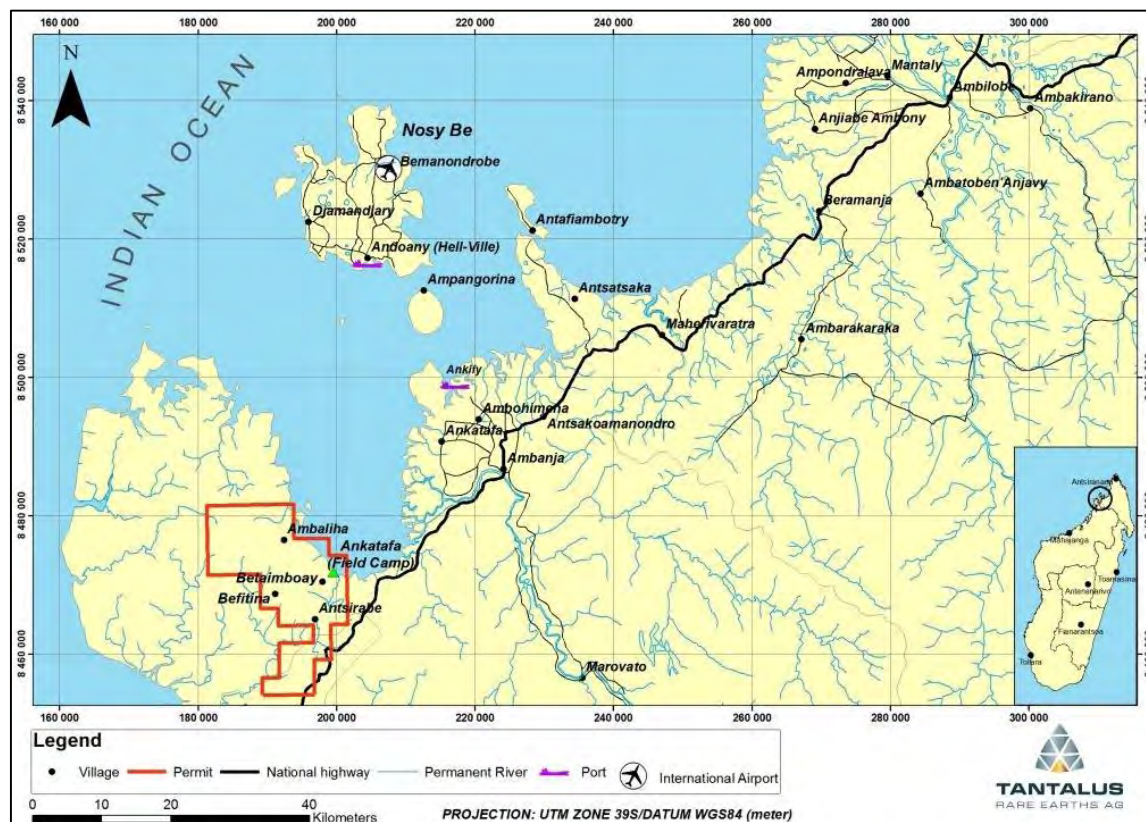


Figure 4-1: Map showing the prospects of the Tantalus Project (TREM, 2014)

The coordinates of the geographic centre (centroid) of the project area in Universal Transverse Mercator (UTM), WGS 84, Zone 38 South are 191457 mE and 8467897 mN. The coordinates in WGS 84, latitude and longitude are -13.8421, 48.1459 and in the local Laborde coordinate system are 584898 (X), 1358752 (Y). The complete sets of coordinates for the TRE project area are provided in Table 4-1.

Table 4-1: Coordinates for the Tantalus project (local Laborde projection)

<b>Point</b>	<b>X</b>	<b>Y</b>	<b>Point</b>	<b>X</b>	<b>Y</b>
<b>A</b>	575005	1362500	L	585000	1355000
<b>B</b>	582500	1362500	M	592500	1355000
<b>C</b>	582500	1357500	N	590000	1355000
<b>D</b>	585000	1357500	O	590000	1352500
<b>E</b>	592500	1365000	P	585000	1352500
<b>F</b>	592500	1367500	Q	585000	1347500
<b>G</b>	587500	1367500	R	582500	1347500
<b>H</b>	587500	1372500	S	582500	1345000
<b>I</b>	575000	1372500	T	590000	1345000
<b>J</b>	595000	1365000	U	590000	1350000
<b>K</b>	595000	1355000	V	592500	1350000

## 4.2 Ownership

Tantalus Rare Earth-AG's subsidiary in Madagascar holds 100% of the mining rights of the project.

## 4.3 Royalties

There are no known royalties on the project.

## 4.4 Permits

The Tantalus project comprises one exploration licence (permit PR 6698) made up of 768 contiguous 625 m by 625 m unit blocks that encompass a total area of 300 km<sup>2</sup>. The permit is currently granted as a "Permis de Recherche" (research permit), or PR, which grants the exclusive right for prospecting and research. The permit is valid until 2017 and can be renewed once more, for a period of three years. See Figure 4-2 below.

The permit was originally held by Calibra Resources and Engineers Madagascar SARL and was subsequently acquired by Zebu Metals Limited in January 2008. Tantalus assumed 100% ownership of the permit in October 2009.



## 4.5 Environmental Liabilities

Tantalus possesses all the required environmental permits to conduct exploration activities on the licence and employ 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 environmental impact study, and an environmental 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 environmental impact study.

The assumptions used to establish the resource estimation cut-off grades included the use of in-situ extraction within areas that have a sufficient topographic slope. This method of extraction is by its nature less invasive than conventional mining. This extraction method is not currently in use in Madagascar, and the authors consider that it is very important to undertake a series of trials to develop the most effective methodology to limit the impact on the environment and gain the social and environmental licenses to operate in this jurisdiction.

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

### 5.1 Physiography

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 5-1) (Gilbertson 2013).



Figure 5-1: 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<sup>2</sup> of the 300 km<sup>2</sup> 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 Tantalus 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 5-2).



**Figure 5-2: Photograph of a general view of the eastern part of the Tantalus project taken in December 2013. First image shows a sector burned to prepare for planting rice with a bamboo stand in the bottom of the valley. Second image is taken from the eastern limit of the caldera showing a patchwork of vegetation from the rotating agriculture usage.**

## 5.2 Accessibility

The nearest international airport to the project area is Fascene, located on the island of Nosy Be (Figure 4-1). Airlines that currently operate include Air Madagascar, Air Austral and Air Italy with destinations including Antananarivo, La Reunion, Johannesburg, Milan and Rome.

Access from Nosy Be to the project area is by boat and Tantalus has its own craft for this purpose. The travel time from 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 4x4 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 intersects the project area in two locations (Figure 4-1). Vehicular access around the project area is limited to a few dirt tracks. These are passable using 4x4 vehicles only and restricted to dry conditions. Most access around the project area is on foot.

## 5.3 Climate

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 Tantalus 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.

## 5.4 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, comprising 4 semi-permanent field camps that has been constructed in strategic locations and include tents, kitchens, generators, bathrooms and a small workshop (Figure 5-3).

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.



Figure 5-3: Photograph of the Ankatafa camp on the coast.

## 5.5 Surface Rights

Tantalus is engaged with the local communities (many small communes) and local land owners to gain access to exploration targets. Tantalus is obligated by law to get permission from landowners prior to doing any exploration work. Tantalus continues to have discussions with local and national governments to establish the framework for continued work on these lands.

## 6 History

### 6.1 Prior Ownership of the Property and Ownership Changes

The license was originally acquired by Calibra Resources and Engineers, in 2003. This was then transferred to Zebu Metals in January 2008, which was acquired by Tantalus in October 2009.

### 6.2 Mineral Exploration Work

#### 6.2.1 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 19<sup>th</sup> 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.

#### 6.2.2 Soviet Exploration

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).

Russian Geologists undertook a programme 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 programme 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 programme, the mapping component extended out from the main study area and covered an area of 10 km<sup>2</sup> 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 6-1. 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 6-1: Exploration completed as part of the Soviet Geological Mission (after OMNIS-SM, 1989; 1990; 1992b; 1992c; 1992d; 1992e)**

<b>Type of geological work</b>	<b>Unit of measure</b>	<b>Planned</b>	<b>Actual</b>
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

## 6.2.3 Contemporary Exploration and Results

### 6.2.3.1 Stream and beach sediment sampling

In 2008, Fugro Consult GmbH (Fugro) was commissioned by the then owners of the Tantalus project area, Zebu Metals Ltd (Zebu) to undertake a week-long reconnaissance field programme. As part of this programme 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.

### 6.2.3.2 Bulk Sampling

In 2008, as part of the Zebu-commissioned programme, 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 6-2.

Table 6-2: Fugro 'Mini bulk sample' results (Fugro, 2008)

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

### 6.2.3.3 Airborne Geophysical Surveys

In 2008, Fugro Airborne Surveys of South Africa completed a helicopter-borne magnetic and radiometric survey. Between the 4<sup>th</sup> and 8<sup>th</sup> 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<sup>2</sup>. 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 findings of the interpretation have been described in Section 5 and complete details are provided in a separate report (Earthmaps Consulting, 2009). Examples of radiometric and magnetic data images are provided in Figure 6-1 and Figure 6-2 respectively.

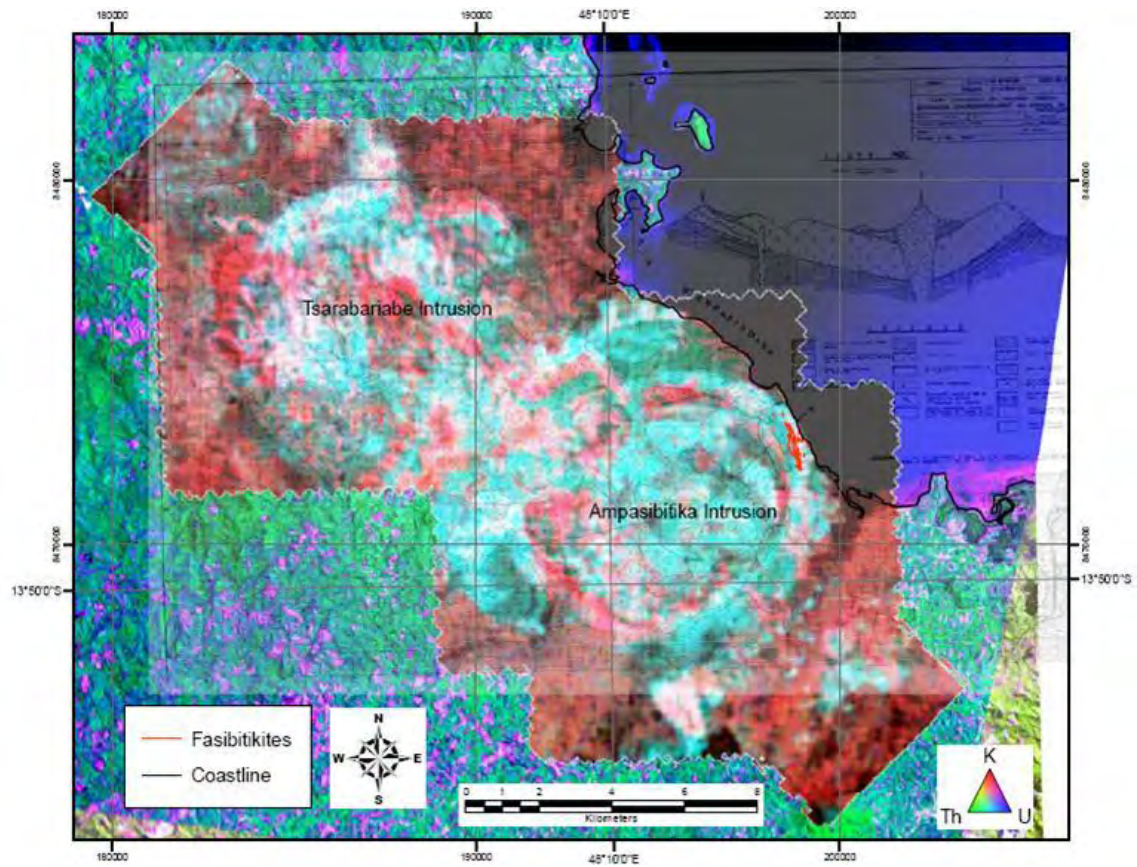


Figure 6-1: Map showing the radiometric Ternary imagery for the Tantalus project (after Earthmaps Consulting, 2009)

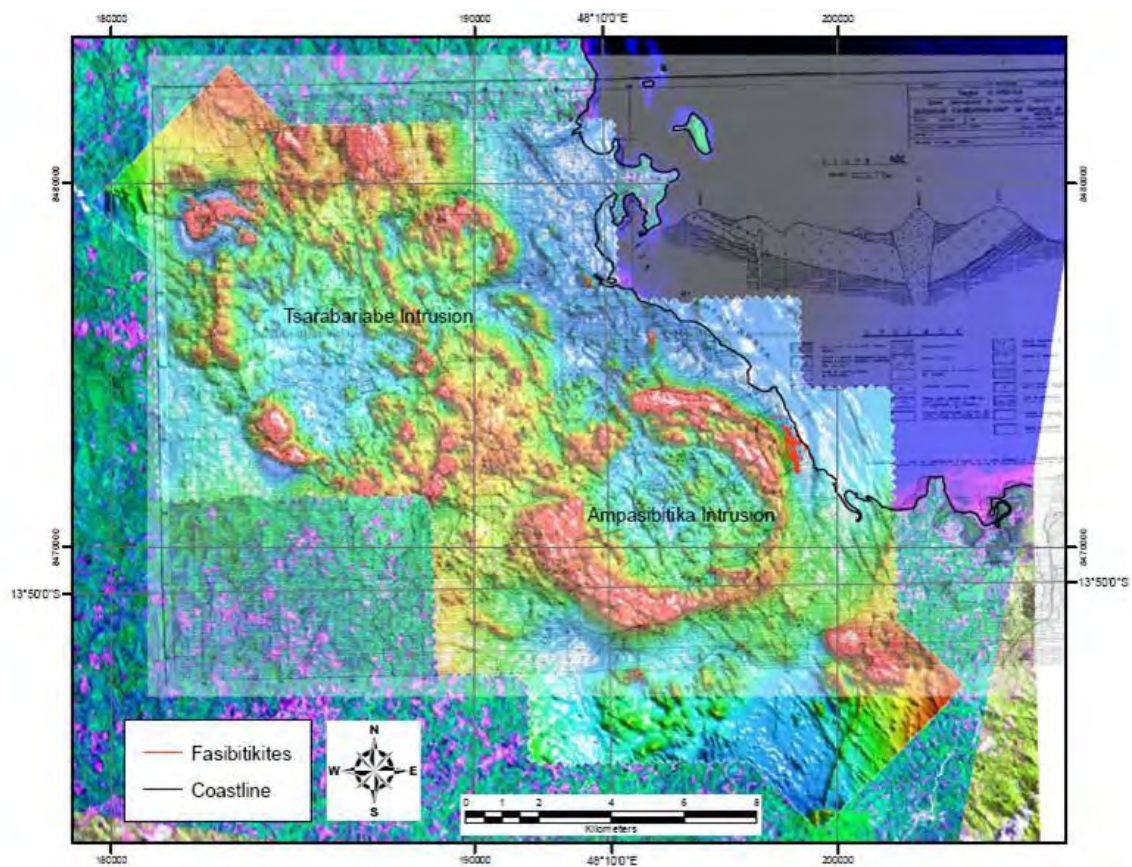


Figure 6-2: Map showing the radiometric Ternary imagery for the Tantalus project (after Earthmaps Consulting, 2009)

## 7 Geological Setting and Mineralization

### 7.1 Regional Geology

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 7-1). 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 Tantalus project area.

### 7.2 Property Geology

#### 7.2.1 Lithology

The Tantalus 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<sup>2</sup>. 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.

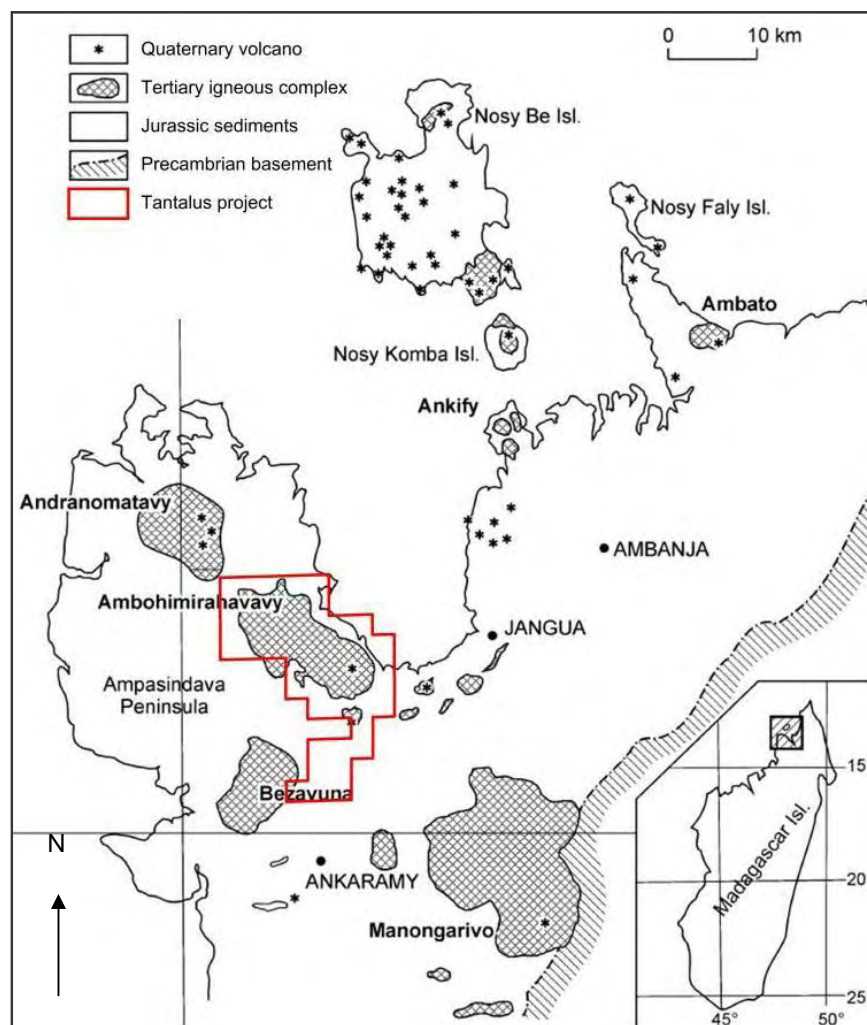


Figure 7-1: Map showing the simplified geological setting of north-western Madagascar (after Ganzeev and Grechishev, 2003)

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 7-2). The published 1:100,000 scale geological map for the Tantalus project area is provided in Figure 7-2.

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).

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 7-4). 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.

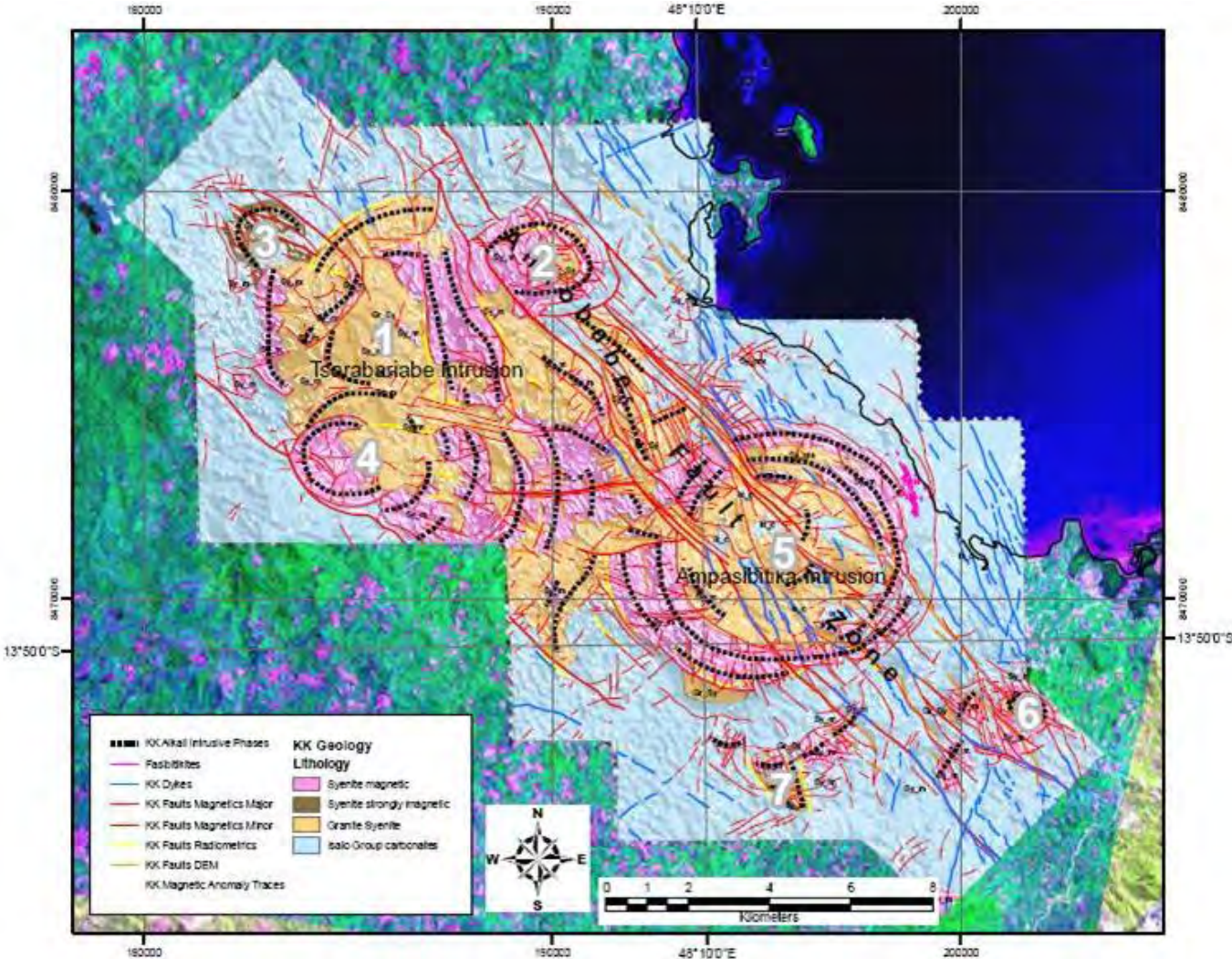


Figure 7-2: Map showing the geological interpretation for the Tantalus project area (after Earthmaps Consulting, 2003)

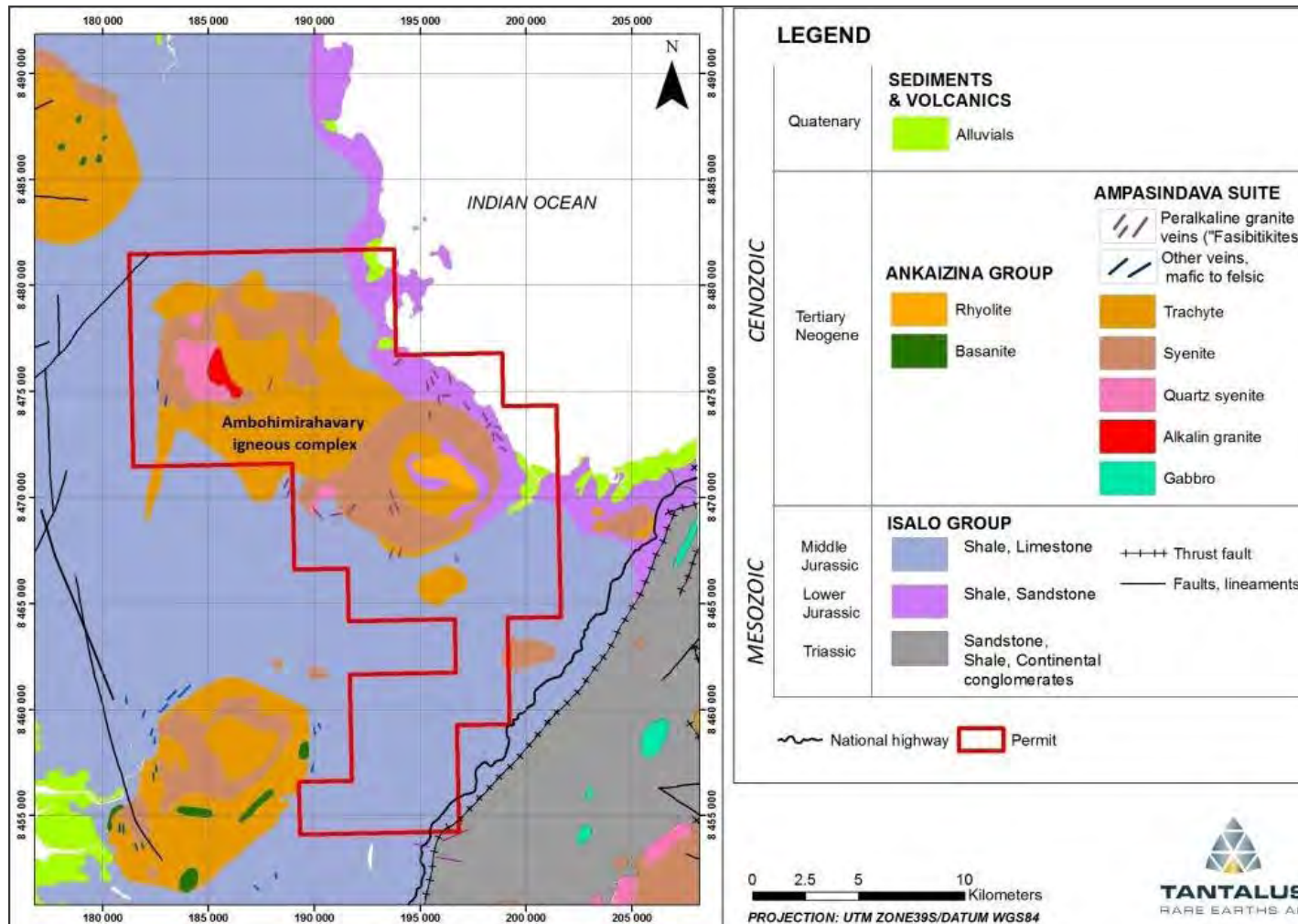


Figure 7-3: Map showing the geological setting for the Tantalus project area (after BGS-USGS, 2008)

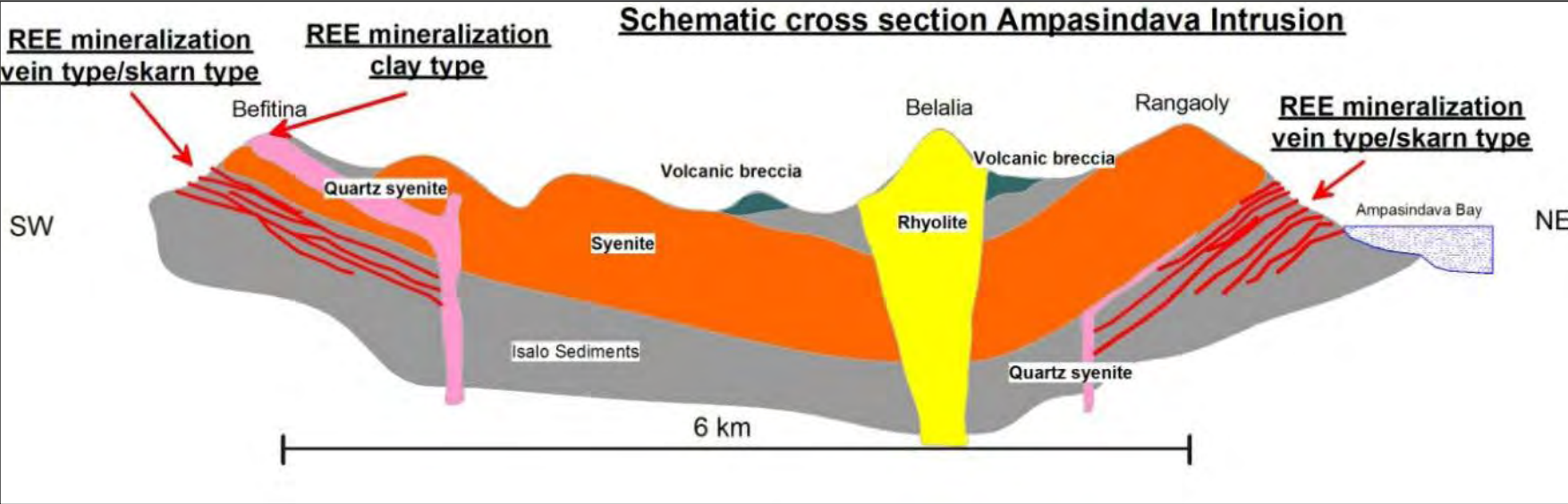


Figure 7-4: Schematic cross-section of the Ambohimirahavy igneous complex (modified from OMNIS-SM, 1992) (Gilbertson, 2013)

## 7.2.2 Structures

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 7-2, Figure 7-5). 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.

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 7-5).

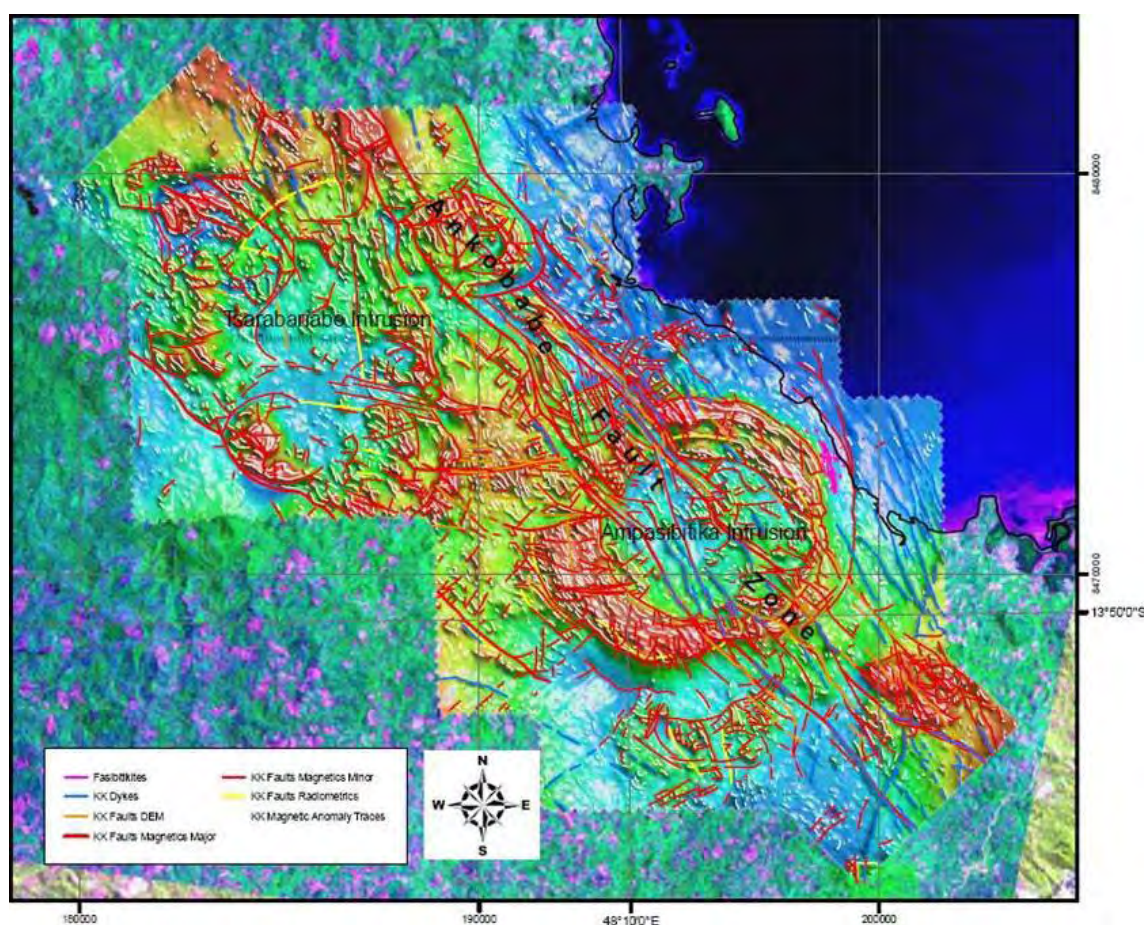


Figure 7-5: Map showing interpreted structural setting for the Tantalus project area (after Earthmaps Consulting, 2009)

### 7.2.3 Regolith

Within the Tantalus 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 Tantalus project area includes a well-developed regolith profile that includes the majority of recognised subdivisions (Figure 7-6).

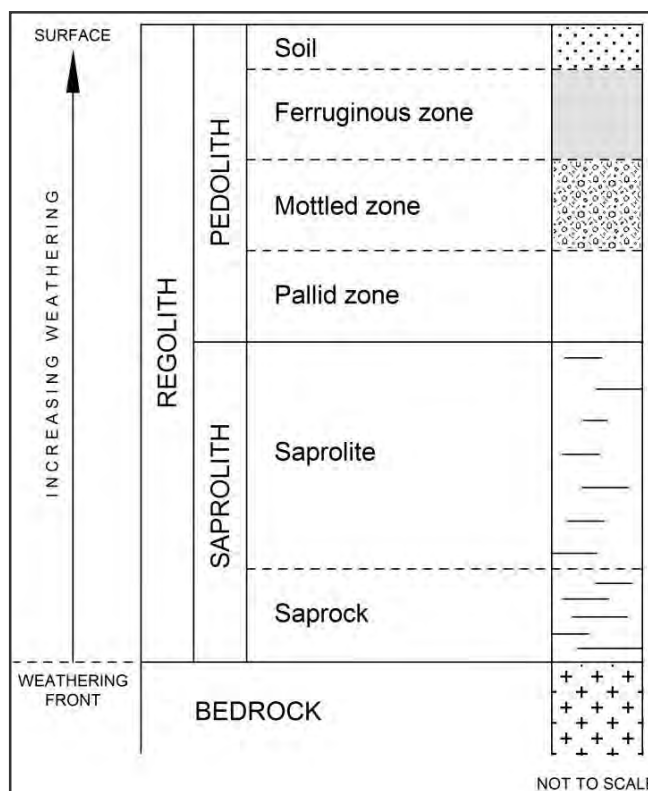


Figure 7-6: Schematic cross-section of the Tantalus project regolith profile (Gilbertson, 2013)

The two primary subdivisions are the pedolith and the saprolith. 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 Tantalus 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 Tantalus 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 Tantalus 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 Tantalus 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 Tantalus 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 saprolith 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 Tantalus 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 Tantalus 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 ( $\text{Fe}_2\text{O}_3$ ), goethite ( $\text{FeO}(\text{OH})$ ) and gibbsite ( $\text{Al}(\text{OH})_3$ ). 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.

### 7.3 Rare Earth Elements and Rare Metals

The rare earth elements (REE), sometimes referred to as the rare earth metals, are a group of 17 chemically similar metallic elements that include the lanthanides, scandium and yttrium (BGS, 2010). 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.

Demand for REEs has increased in recent years because of their wide and diverse use in high-technology applications. However, the global production and supply of REEs comes from only a few sources with China producing more than 95% of the World's supply (USGS, 2011). Because of China's decision to restrict exports of REEs, industrialised countries are concerned about supply shortages and REE prices reached a peak in the summer of 2011. Since 2011 prices have declined significantly, and have stabilized above historical levels.

The Tantalus deposit includes appreciable amounts of critical rare earths Dysprosium, Europium, Neodymium, Terbium, and Yttrium as defined by the U.S. Department of Energy. In the 2011 Critical Materials Report from the U.S. Department of Energy, it was revealed that five of the 16 rare earth elements were found to be critical in terms of supply risk in the short term (present to 2015) and into the medium term (2015 – 2025). These elements are termed "critical" because of their scarcity (more on this in Section 8.4).

Specific applications for the individual rare earth elements are provided in Table 7-1

Table 7-1: Rare Earth Elements and their uses.

	<i>Element</i>	<i>Symbol</i>	<i>Atomic No.</i>	<i>Usages</i>
<b>Light REE</b>	Lanthanum	La	57	Glass, flint, hydrogen storage, batteries, camera lenses, catalysts
	Cerium	Ce	58	Polishing, glass/ceramic colouration, catalysts
	Praseodymium	Pr	59	Magnets, lasers, lighting, glass/ceramic colouration, flint
	Neodymium	Nd	60	Magnets, lasers, glass/ceramic colouration, ceramic capacitors
<b>Heavy REE</b>	Samarium	Sm	62	Magnets, lasers, neutron capture, masers
	Europium	Eu	63	Red and blue phosphors (TV colour), lasers, mercury-vapour lamps
	Gadolinium	Gd	64	Magnets, glass, lasers, x-ray tubes, computer memory, neutron capture
	Terbium	Tb	65	Green phosphors, lasers, fluorescent lamps
	Dysprosium	Dy	66	Magnets, lasers
	Holmium	Ho	67	Lasers
	Erbium	Er	68	Lasers, vanadium steel
	Thulium	Tm	69	Portable x-ray machines
	Ytterbium	Yb	70	Infrared lasers, reducing agent
	Lutetium	Lu	71	PET Scan detectors, glass
	Yttrium	Y	39	Lasers, superconductors, microwave filters

Other rare or speciality metals include the likes of tantalum (Ta), niobium (Nb), zirconium (Zr), hafnium (Hf) and gallium (Ga), all of which occur in the Tantalus project area and are important for high-technology applications.

## 7.4 Mineralization

### 7.4.1 Introduction

The Tantalus 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.

### 7.4.2 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 feldspathoids, 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.  $\text{Na}_2\text{O} + \text{K}_2\text{O} > \text{Al}_2\text{O}_3$ . 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 that 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 is 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  $\mu$ r/h to 1,450  $\mu$ r/h (averaging 300  $\mu$ r/h), relative to an estimated background of 25  $\mu$ r/h to 40  $\mu$ r/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.

### 7.4.3 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 are 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).

#### 7.4.3.1 *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.

#### 7.4.3.2 *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).

#### 7.4.3.3 *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).

#### 7.4.3.4 *Summary*

The regolith material in the Tantalus 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 Tantalus 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.

## 7.5 Mineralisation Model

Tantalus is focussed on the exploration, delineation and ultimately exploitation of regolith-hosted REE mineralisation analogous to the ion adsorption REE mineralisation found in China. Please see section 8 for a detailed description of the deposit model and characteristics. The fersite 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.

## 7.6 Exploration Characteristics

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 programmes 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).

As 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 Tantalus 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.

## 8 Deposit Types

### 8.1 Deposit Summary

The Tantalus project area is underlain by the Ambohimirahavavy igneous complex that encompasses an area of approximately 150 km<sup>2</sup>. 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.

### 8.2 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.

### 8.3 Mineralisation Characteristics

Geometrically the regolith profile in the Tantalus 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 saprolith. 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 saprolith 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 Tantalus project area.

### 8.4 Critical Rare Earth Elements

The Tantalus Ionic clay project displays a pervasive and well-balanced rare earth distribution that is prevalent throughout the deposit. The deposit includes appreciable amounts of critical rare earths Dysprosium, Europium, Neodymium, Terbium, and Yttrium as defined by the U.S. Department of Energy. In the 2011 Critical Materials Report from the U.S. Department of Energy, it was revealed that five of the 16 rare earth elements were found to be critical in terms of supply risk in the short term (present to 2015) and into the medium term (2015 – 2025). 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 are: Dysprosium, Europium, Neodymium, Terbium, and Yttrium. These five rare earth elements are used in magnets for wind turbines and electric vehicles or phosphors in energy-efficient lighting. Since publication of the 2013 report we have seen a significant drop in the prices of all REE, however the Critical REE have retained their value better than their “non-critical” counterparts.

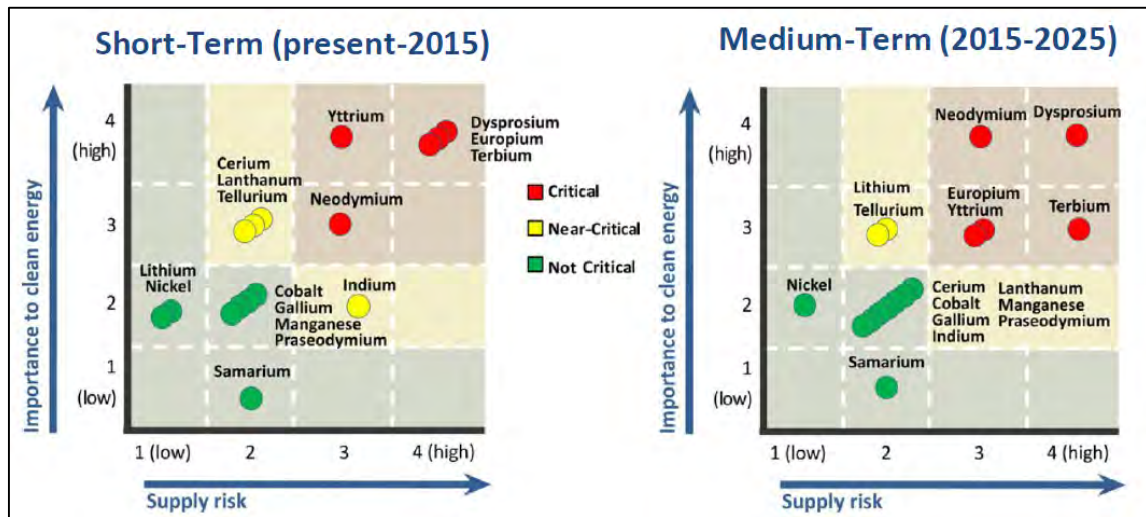


Figure 8-1: Short and Medium term Criticality Matrix

## 9 Exploration

### 9.1 Outcrop Sampling

Tantalus has 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 9-1 and are summarised by prospect in Table 9-1.

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).

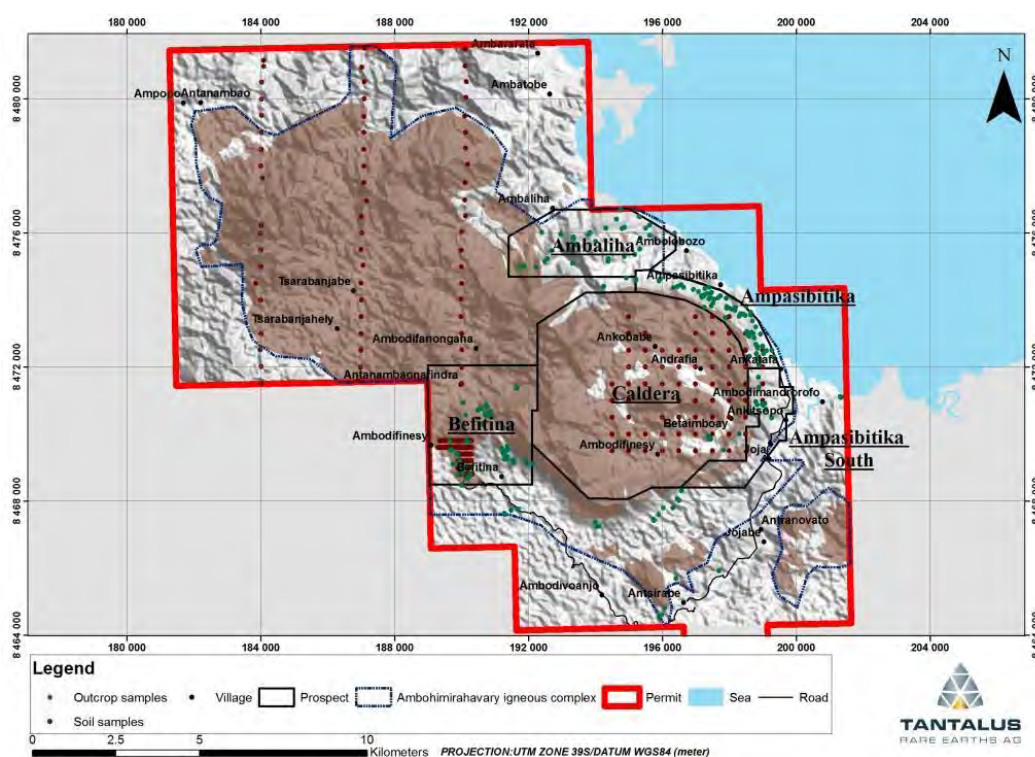


Figure 9-1: Map showing the locations of the Tantalus outcrop and soil samples.

Table 9-1: Summary of the Tantalus outcrop sample results (Gilbertson, 2013)

<b>PROSPECT</b>	<b>STATS</b>	<b>TREO (ppm)</b>	<b>HREO (%)</b>	<b>Nb<sub>2</sub>O<sub>5</sub> (ppm)</b>	<b>Ta<sub>2</sub>O<sub>5</sub> (ppm)</b>	<b>ZrO<sub>2</sub> (ppm)</b>	<b>HfO<sub>2</sub> (ppm)</b>	<b>Ga (ppm)</b>
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						

## 9.2 Soil Sampling

Tantalus has completed soil sampling in several parts of the Tantalus 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.



predominantly regolith (pedolith and saprolith), but also bedrock in a few places. Trench TANT5 intersected only saprolith. 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 9-2.

Table 9-2: Summary of the Tantalus trench sample results (Gilbertson, 2013).

<b>PROSPECT</b>	<b>STATS</b>	<b>TREO (ppm)</b>	<b>HREO (%)</b>	<b>Nb<sub>2</sub>O<sub>5</sub> (ppm)</b>	<b>Ta<sub>2</sub>O<sub>5</sub> (ppm)</b>	<b>ZrO<sub>2</sub> (ppm)</b>	<b>HfO<sub>2</sub> (ppm)</b>	<b>Ga (ppm)</b>
<b>TANT2 (Befitina)</b>	MIN	899	16	383	28	1,932	50	48
	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						
<b>TANT3 (Befitina)</b>	MIN	661	13	192	12	1,299	30	49
	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						
<b>TANT4 (Caldera)</b>	MIN	467	11	131	9	401	12	30
	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						
<b>TANT5 (Caldera)</b>	MIN	470	16	183	11	655	17	32
	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 Caldera 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.

## 9.4 Pitting

To date, Tantalus has manually excavated a total of 4474 pits for the purposes of assessing regolith-hosted REE mineralisation (Figure 9-3). 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.

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. 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 9-3 and the summary statistics for the pits excavated to date are provided in Table 9-3.



Figure 9-3: Photos of work undertaken in the field. 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.

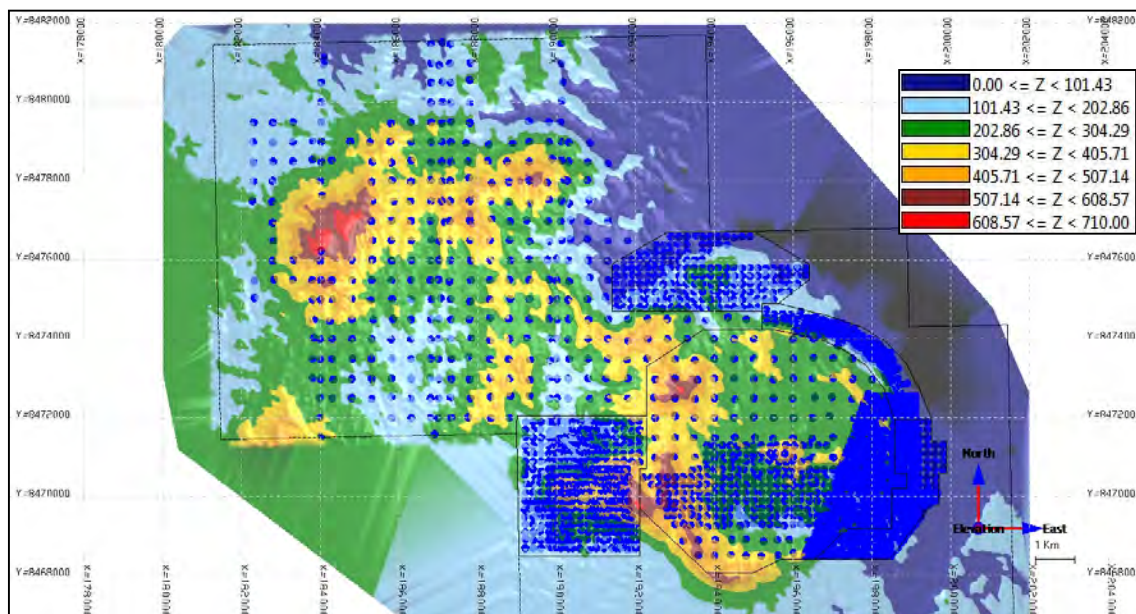


Figure 9-4: Topographic image showing the distribution of pits within the TRE project. Areas that are completely covered in points are covered by a grid approaching 50m. The most sparsely drilled/pitted area have a grid spacing of around 250m. Hotter colors indicate higher elevation.

Table 9-3: Summary of analyses from the Tantalus pitting campaigns.

Layer*	Statistic	TREO (ppm)	TREOnoCe (ppm)	LREO (ppm)	HREO (ppm)	Nb <sub>2</sub> O <sub>5</sub> (ppm)	Ta <sub>2</sub> O <sub>5</sub> (ppm)	ThO <sub>2</sub> (ppm)	U <sub>3</sub> O <sub>8</sub> (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 TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

### 9.4.1 Pit Logging and Sampling Procedures

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. 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 at Ankatafa then transferred by boat and / or by 4x4 vehicle to the Tantalus sample preparation facility in Ambanja.

The density measurement technique was improved in 2013 to ensure that the sample humidity could be established reliably. A ledge was carved into the wall of the pit on the same day that its depth was excavated (Figure 9-3c). A tube is carefully hammered into the ledge to fill the tube. The tube is dislodged from the wall and excess material is carefully shaved off with a machete. The regolith is carefully emptied into a sample bag and weighed. This bag is carefully sealed and is taped to the assay sample bag from the same depth. Once the sample arrives at the Tantalus preparation lab the density sample is weighed once more to validate the humid weight before drying. As the density sample is dried in ovens, it is weighed at regular intervals to ensure that the sample is completely dry without the causing dehydration of mineralogical volatiles.

## 9.5 Window Sampling

To date, Tantalus has drilled a total of 47 window sampling holes using a Geotools Wacker BH23 unit ([www.ngdgeo.de/index.php/wacker-bh-23.html](http://www.ngdgeo.de/index.php/wacker-bh-23.html)). 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 (Figure 9-5).

The window sampling programme has 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 9-5: Window sampler demonstration. 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.

## 9.6 Exploration conclusions

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

## 10 Drilling

### 10.1 Core Drilling

The initial 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 10-1.



Figure 10-1: Photograph of the Versadrill Kmb.4km drill rig in operation (Gilbertson, 2013)

A total of 277 holes were drilled in the Ampasibitika prospect, 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 9-2.

The drilling programme 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  $-70^{\circ}$  with azimuths to the east and northeast, but also included holes drilled at  $-45^{\circ}$  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 6-4. 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.

## 10.2 Core Logging and Sampling Procedures

Once drilled the core was placed in wooden core boxes by the off-siders and wooden depth markers inserted. Prior to being manually transported from the drill-site to either the field camp at Ankatafa or Ampasibitika village, the boxes were sealed with a plywood lid to prevent core displacement. From either of these localities the core was then transferred by boat and / or by 4x4 vehicle to the Tantalus sample preparation facility in Ambanja.

Once at the sample preparation facility the core was logged and photographed by Tantalus geologists and marked-up for sampling. The initial sampling strategy was restricted to intersections that included intrusive rocks, were radioactive and / or fluoresced under ultraviolet light. This resulted in incomplete sampling of the bedrock sections of the drillholes that was subsequently remedied by selective infill sampling.

## 10.3 Core Drilling Summary

The holes drilled in the Ampasibitika prospect intersected between 0.0 m and 41.35 m of regolith (corrected thickness) that averages 13.8 m thick. The underlying bedrock comprised the expected sedimentary package of mudstones interbedded with sandstones, marls and minor limestone that have been intruded by alkaline and peralkaline intrusive rocks. However, what was intersected was not in keeping with the simplistic “ring-dyke” interpretation illustrated in the geological cross-section in Figure 7-6. Rather than being a series of thick intrusives with consistent orientations and predictable continuity, they were observed to have highly variable thicknesses and much more complex geometries. Furthermore, there were often wide intersections of sedimentary rocks devoid of any intrusives. Despite the erratic nature of the intrusives in the Ampasibitika prospect area, the drilling programme did confirm the presence of mineralised rocks with grades of up to 23,857 ppm / 2.39% TREO.

Ultimately the drilling programme at the Ampasibitika prospect has established that the continuity, predictability and grades of the mineralised intrusives are insufficient for a bedrock-hosted REE resource. For this reason it was decided to focus exploration efforts on the regolith-hosted mineralisation.

## 11 Sample Preparation, Analyses and Security

### 11.1 Sample Quality Assurance and Quality Control (QAQC)

Tantalus 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.

#### 11.1.1 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 within the TRE project and have sensible grade values; unfortunately none of these standards were validated through a round robin and statistical evaluation. Two new standards have been prepared by commercial lab and sent for round robin; these new standards will be ready for the next exploration season.

However, 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 (similar to Figure 11-1 to Figure 11-9) 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 11-1 summarises the standards test results while Figure 11-1 to Figure 11-9 displays the information graphically for a selection of relevant elements. 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 11-1: Summary of the statistical analysis for REE at the Tantalus Project

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%

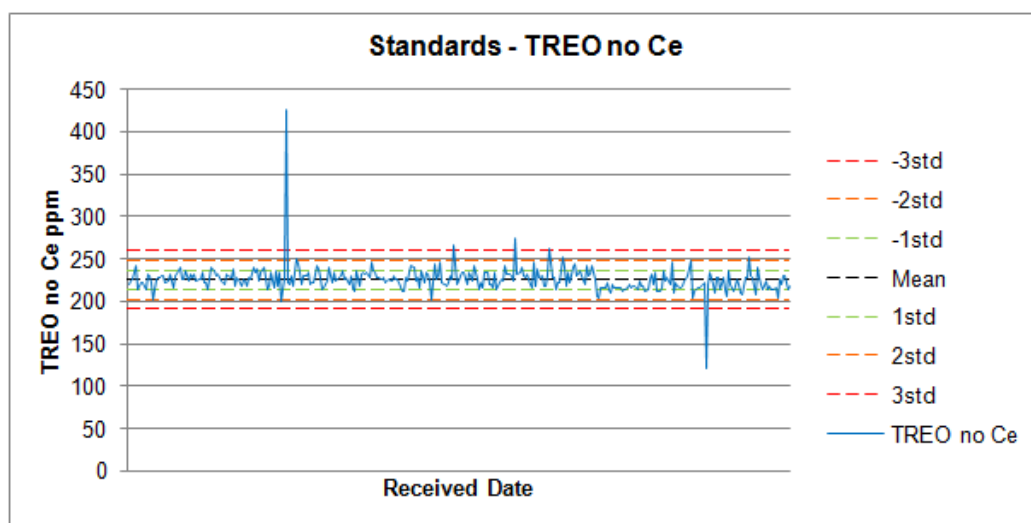


Figure 11-1: Tantalus Standard – TREO

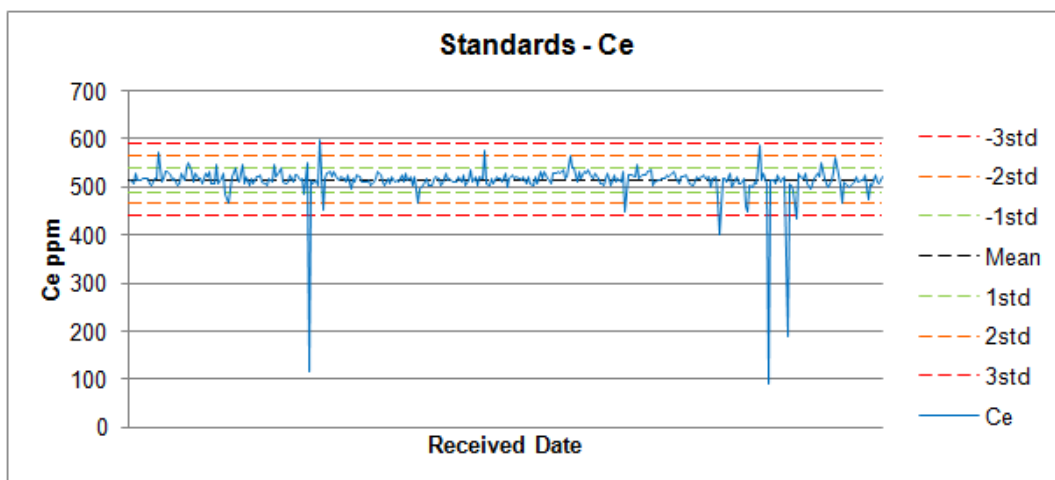


Figure 11-2: Tantalus Standard – Ce

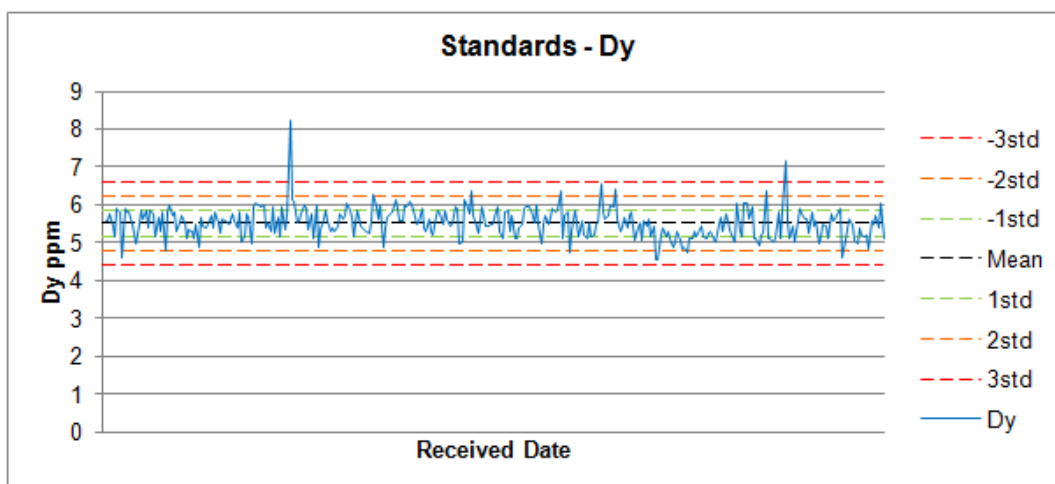


Figure 11-3: Tantalus Standard – Dy

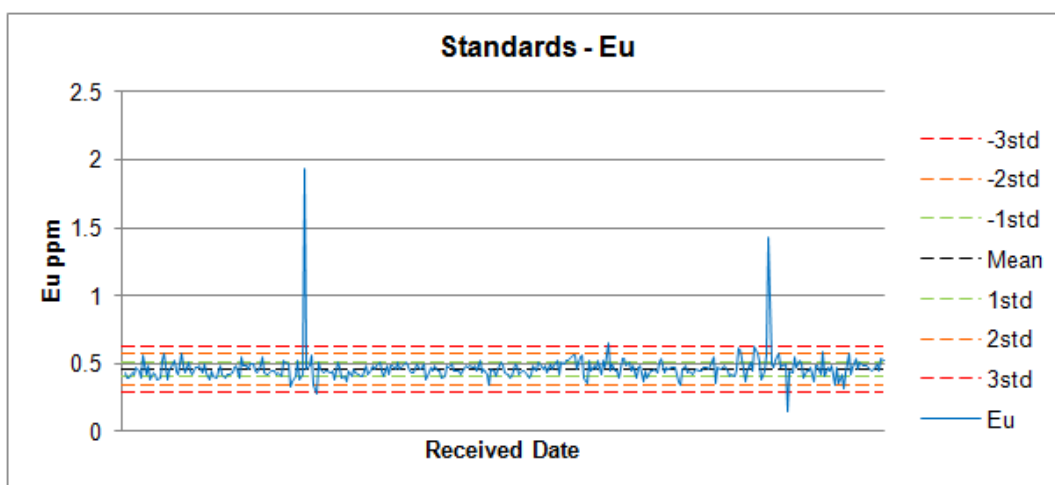


Figure 11-4: Tantalus Standard – Eu

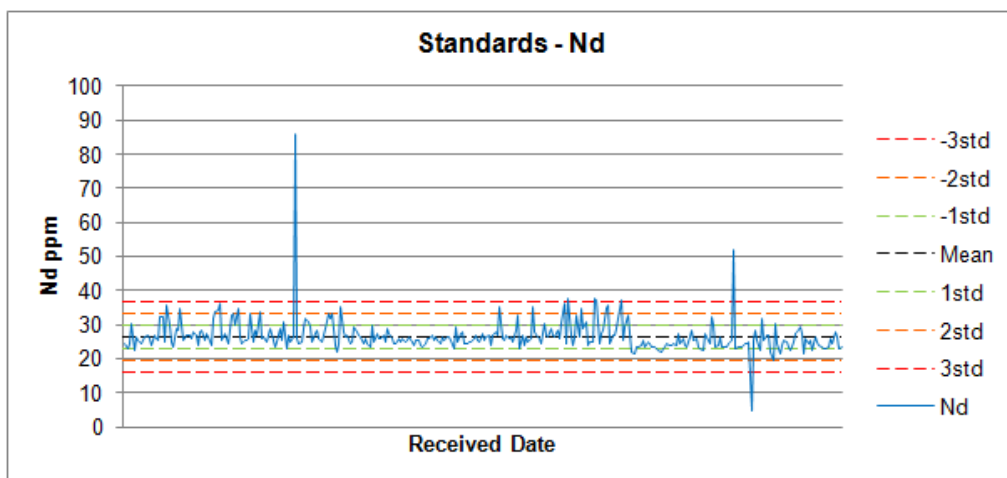


Figure 11-5: Tantalus Standard – Nd

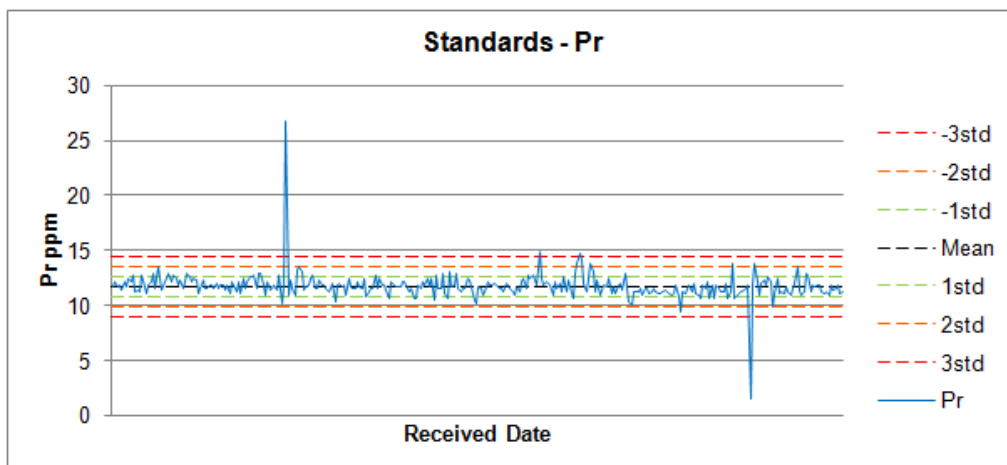


Figure 11-6: Tantalus Standard – Pr

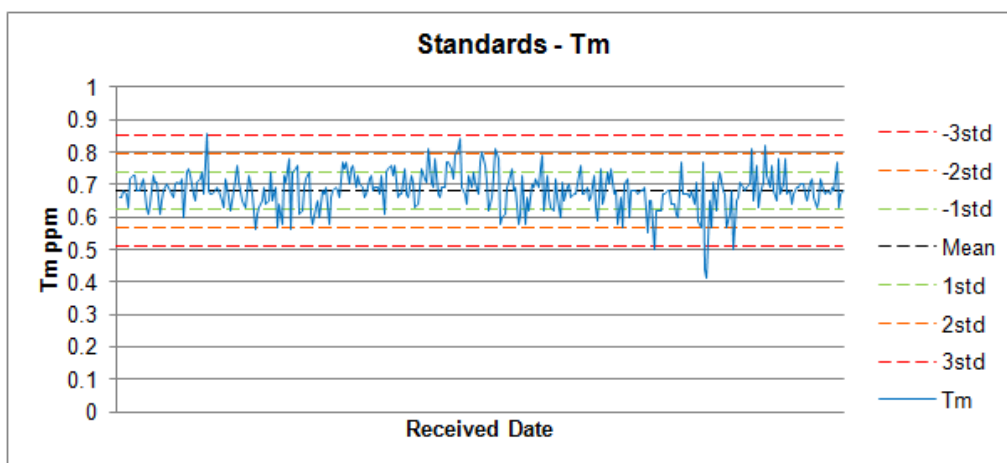


Figure 11-7: Tantalus Standard – Tm

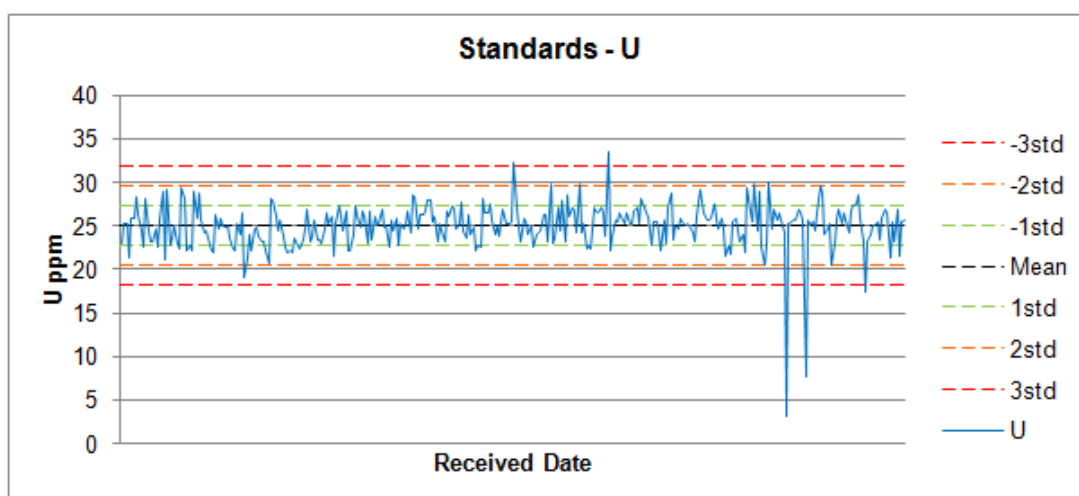


Figure 11-8: Tantalus Standard – U

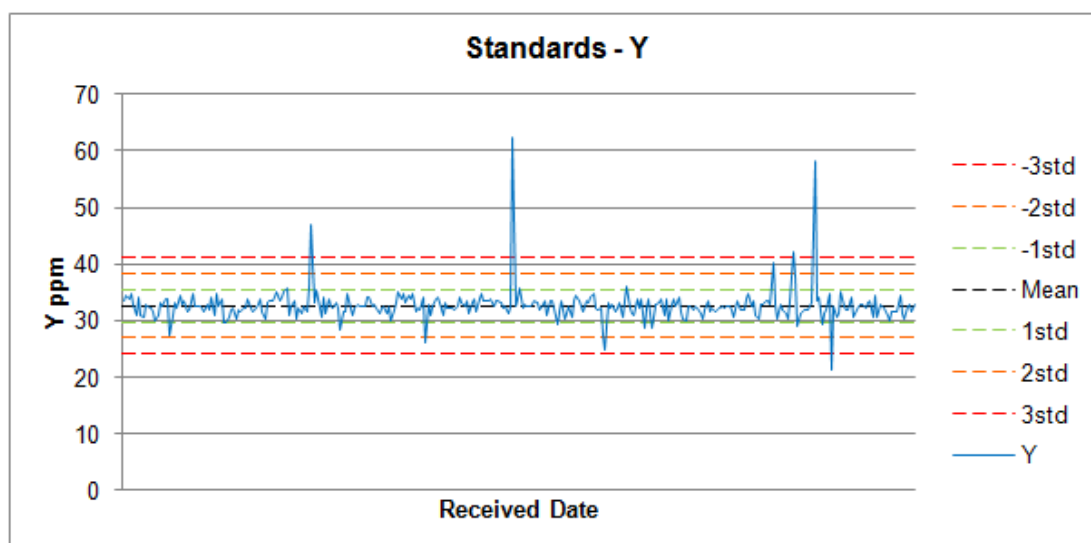


Figure 11-9: Tantalus Standard – Y

### 11.1.2 Blanks

Blank material is inserted into the sample stream in order to assess any sample contamination. Tantalus inserts blank mudstone material collected from a quarry in mainland Madagascar that is known to be devoid of REE mineralisation.

Figure 11-10 to Figure 11-20 display the results of the blank material. 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.

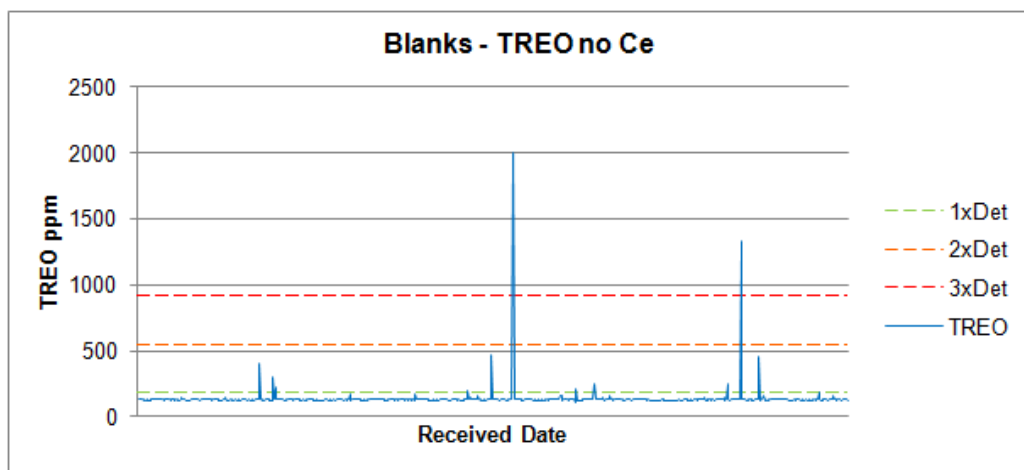


Figure 11-10: Tantalus Blank – TREO

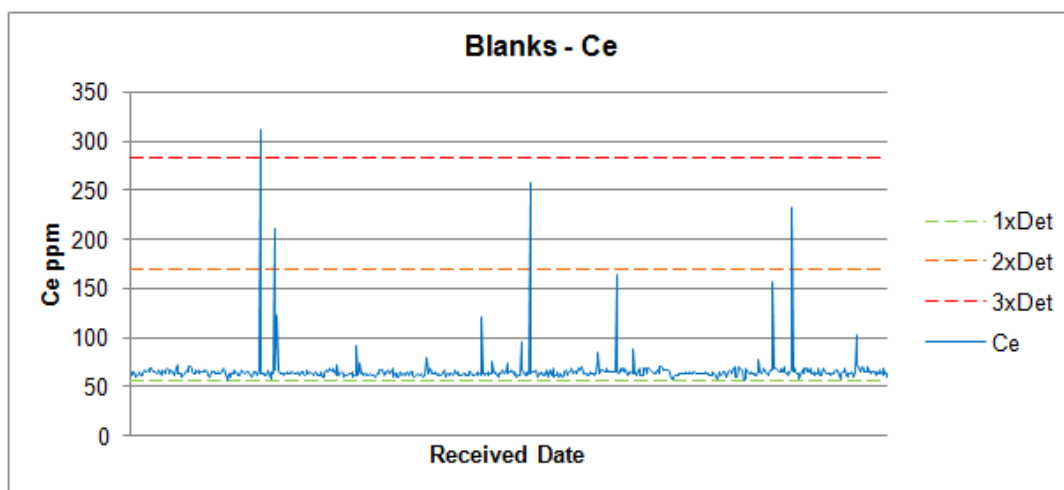


Figure 11-11: Tantalus Blank – Ce

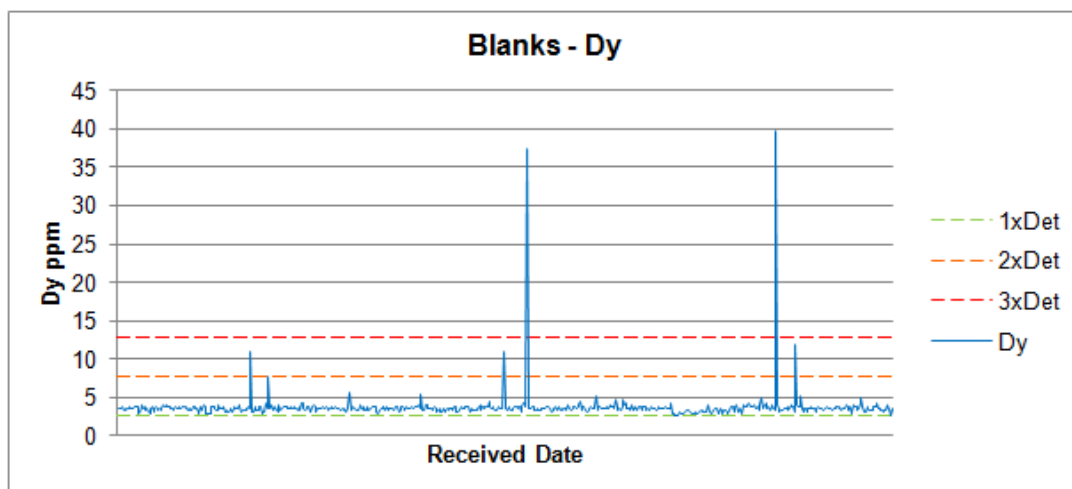


Figure 11-12: Tantalus Blank – Dy

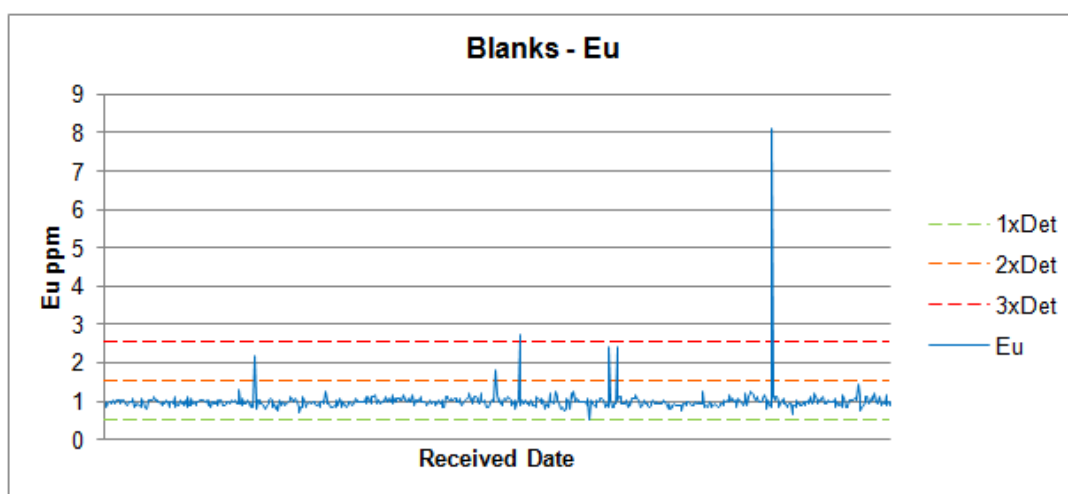


Figure 11-13: Tantalus Blank – Eu

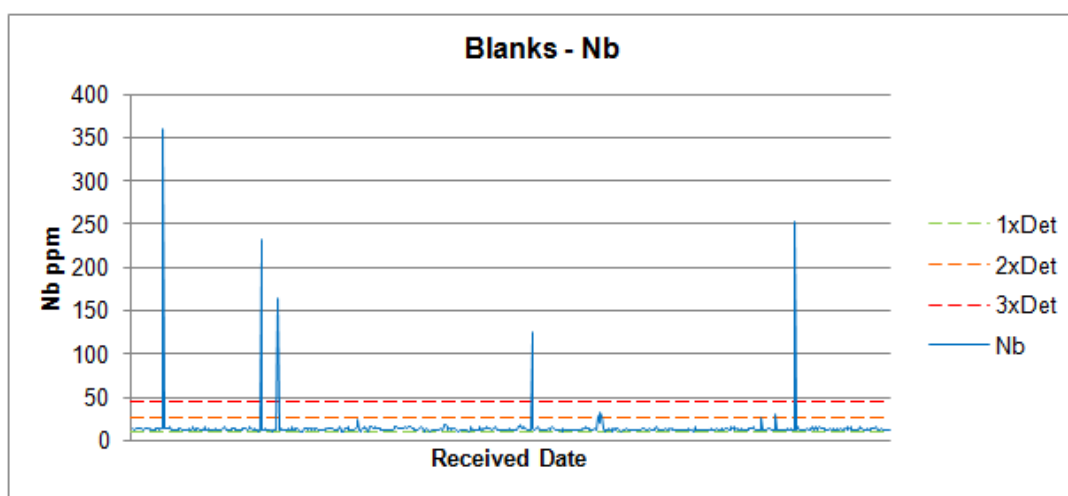


Figure 11-14: Tantalus Blank – Nb

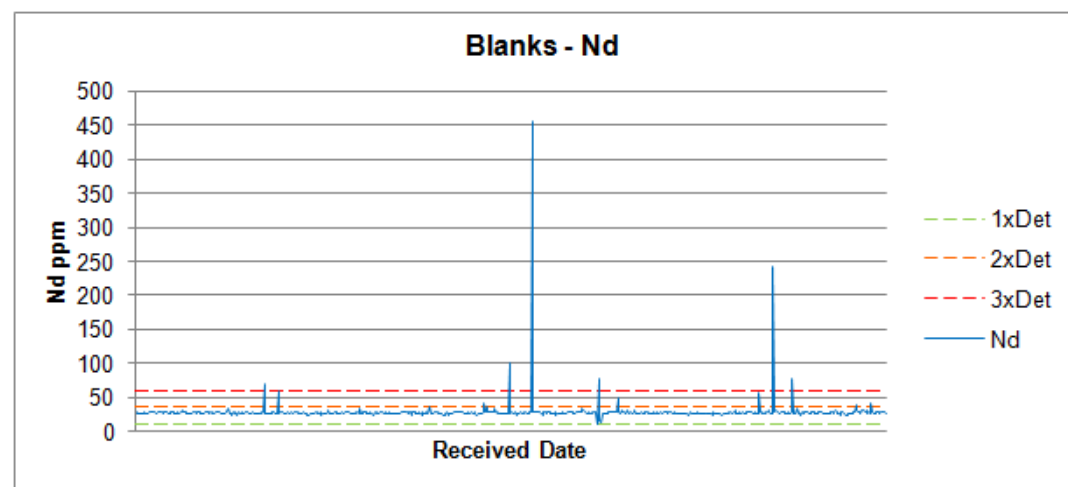


Figure 11-15: Tantalus Blank – Nd

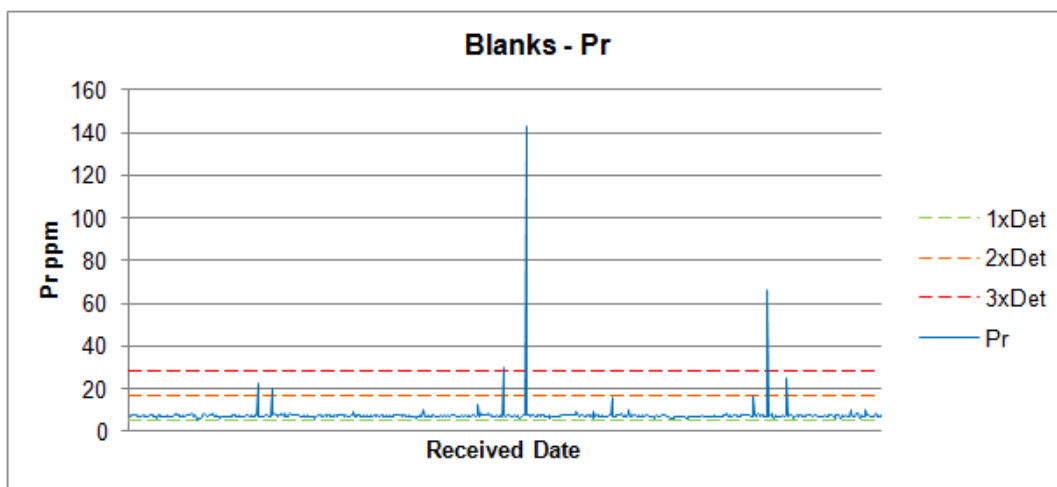


Figure 11-16: Tantalus Blank – Pr

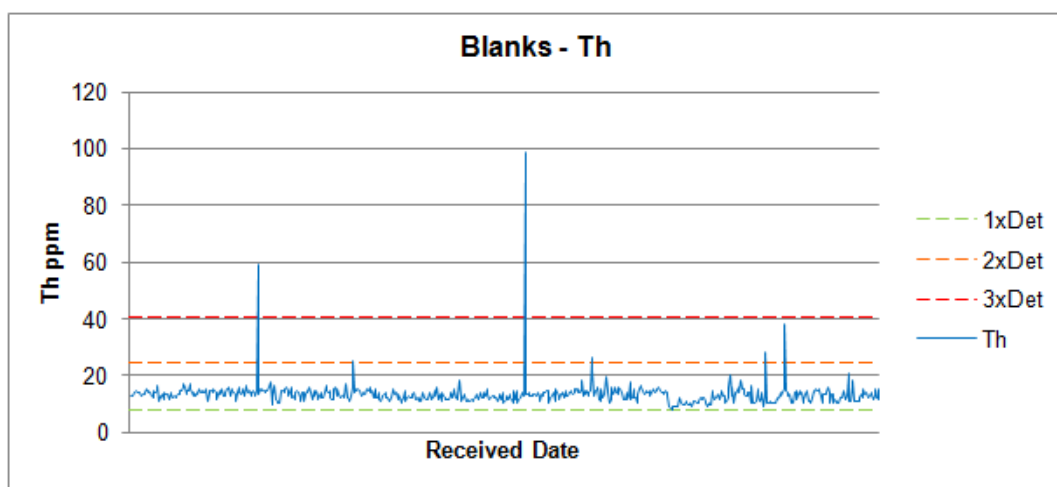


Figure 11-17: Tantalus Blank – Th

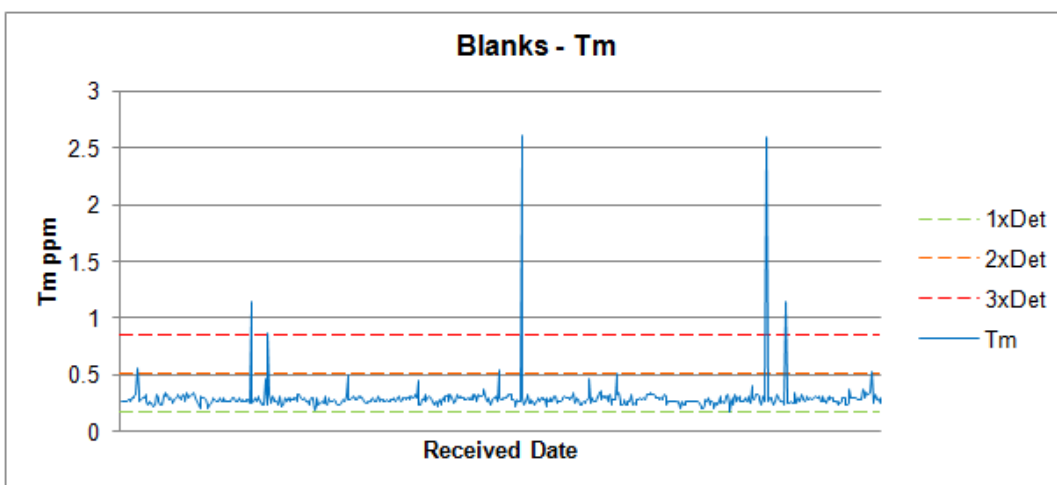


Figure 11-18: Tantalus Blank – Tm

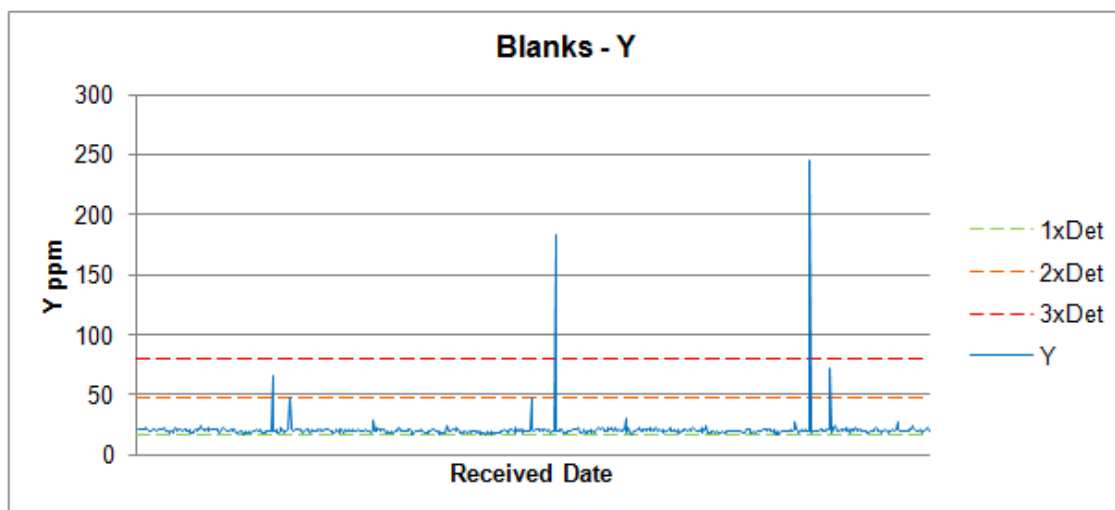


Figure 11-19: Tantalus Blank – Y

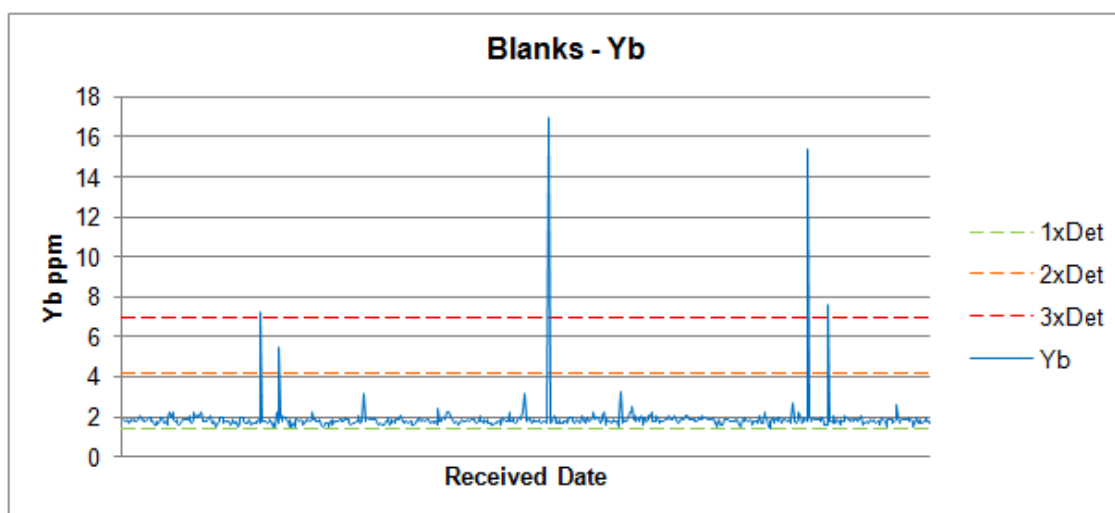


Figure 11-20: Tantalus Blank – Yb

### 11.1.3 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) are 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 Figure 11-21 to Figure 11-31, with the entire sample shown on the left and a zoomed in graph on the right. 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

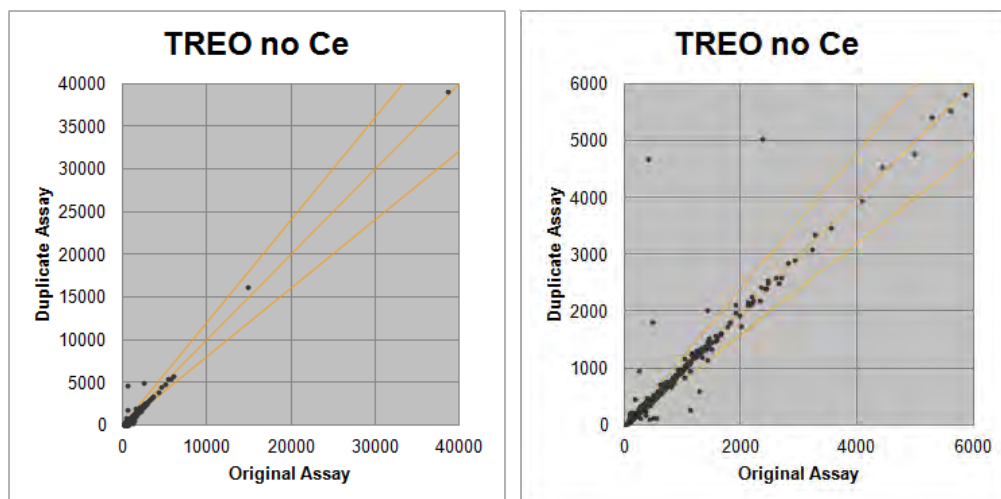


Figure 11-21: Original v Duplicate TREO (ppm) assays.

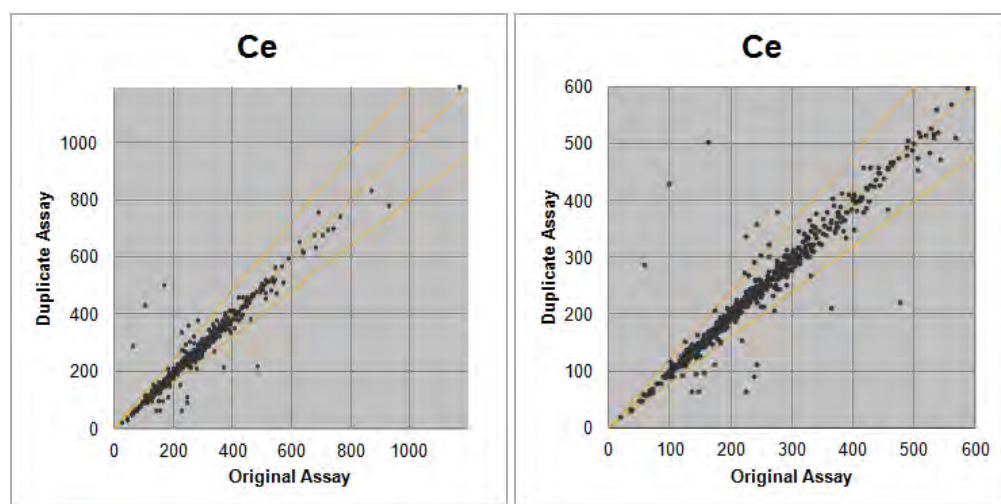


Figure 11-22: Original v Duplicate Ce (ppm) assays.

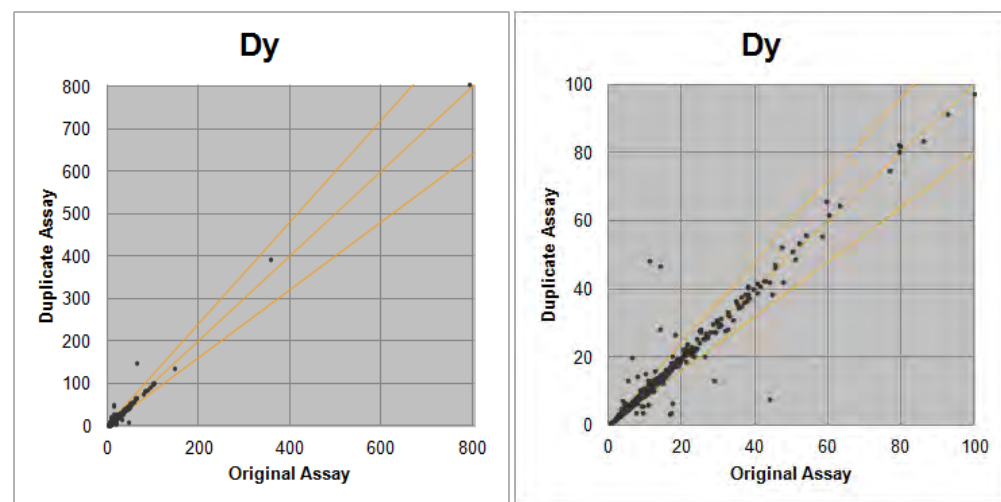


Figure 11-23: Original v Duplicate Dy (ppm) assays.

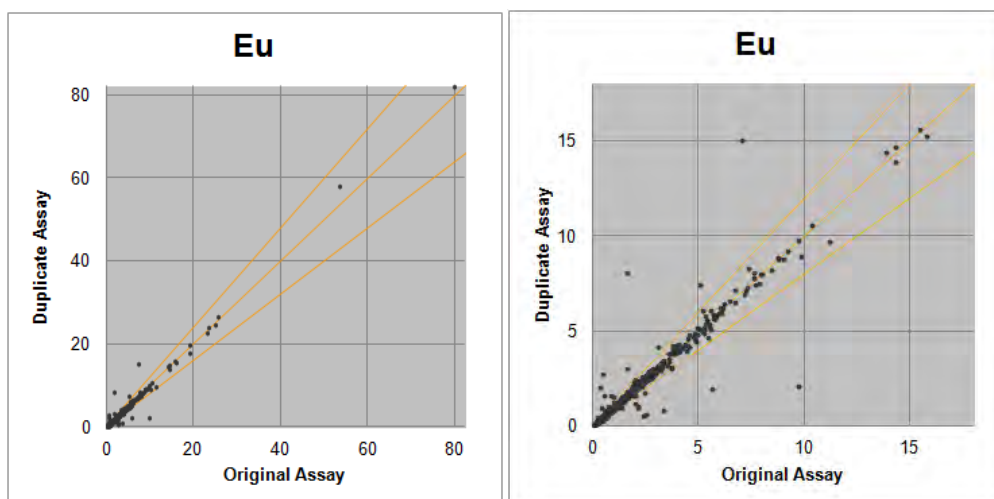


Figure 11-24: Original v Duplicate Eu (ppm) assays.

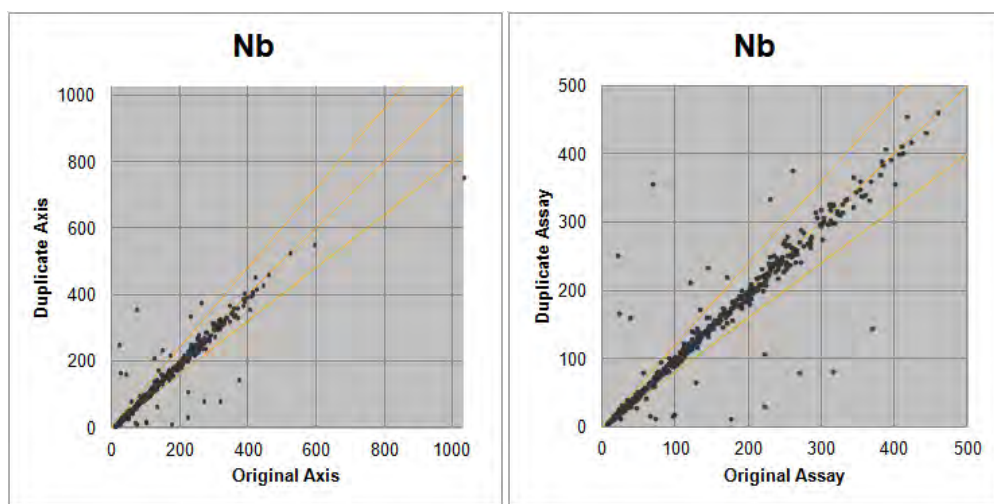


Figure 11-25: Original v Duplicate Nb (ppm) assays.

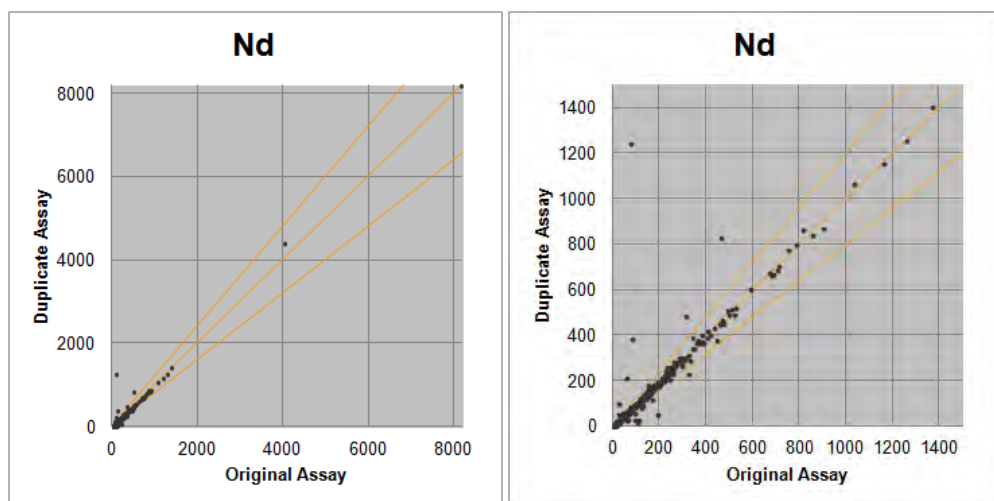


Figure 11-26: Original v Duplicate Nd (ppm) assays.

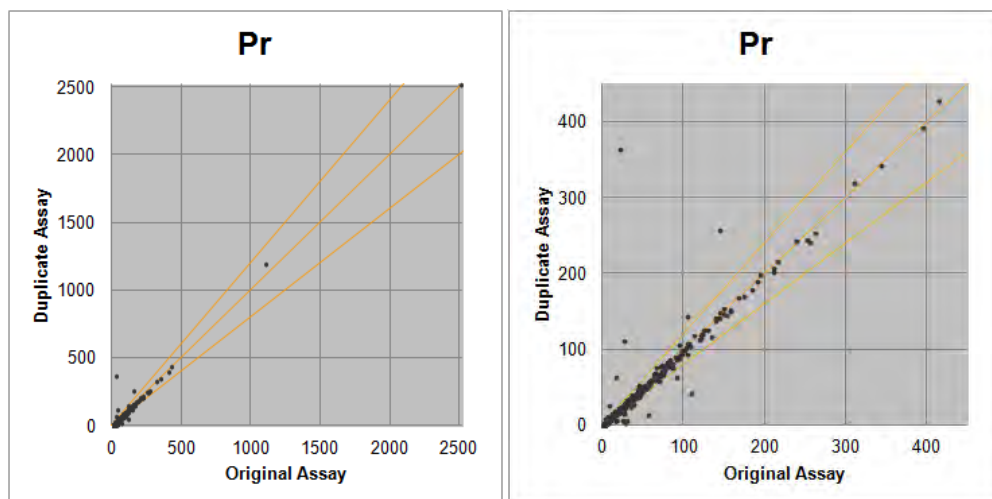


Figure 11-27: Original v Duplicate Pr (ppm) assays.

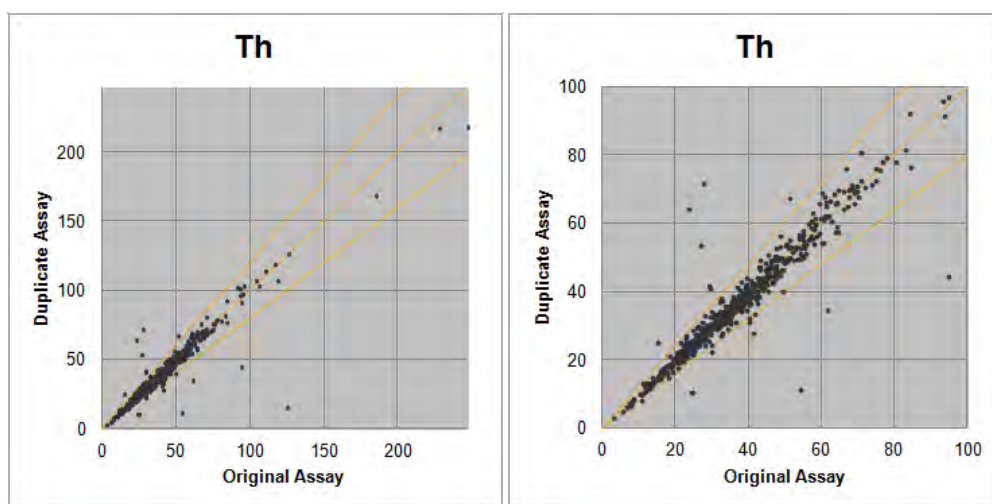


Figure 11-28: Original v Duplicate Th (ppm) assays.

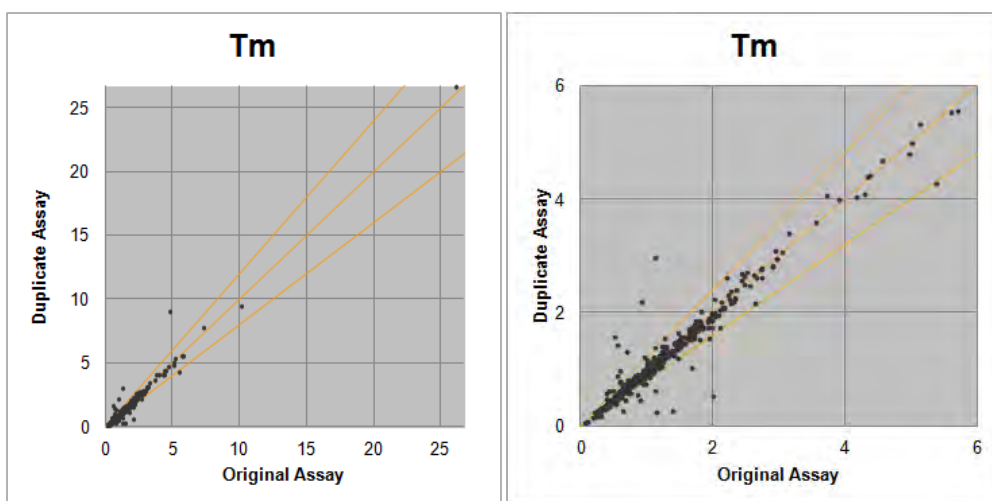


Figure 11-29: Original v Duplicate Tm (ppm) assays.

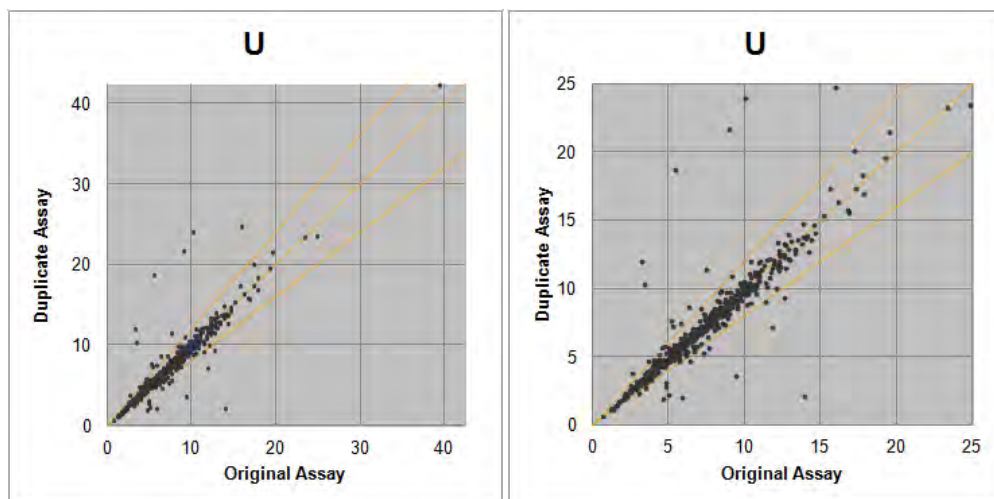


Figure 11-30: Original v Duplicate U (ppm) assays.

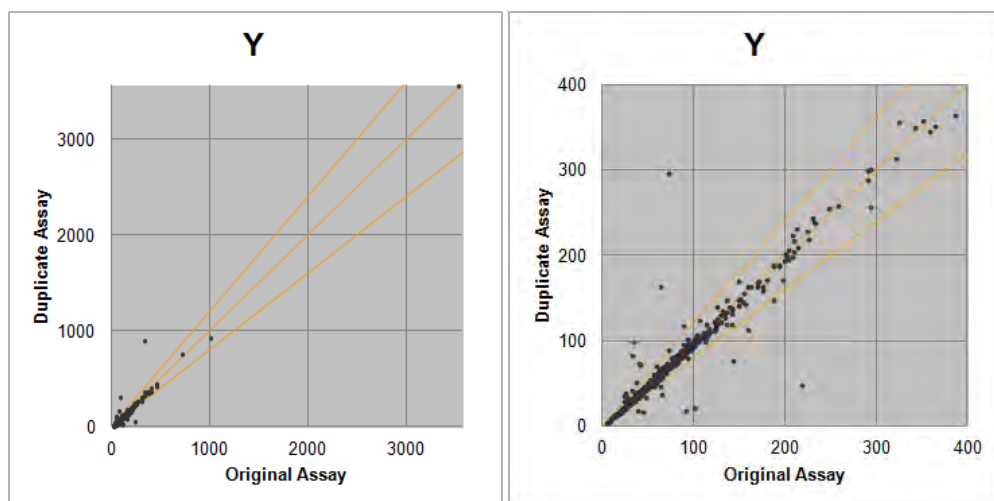


Figure 11-31: Original v Duplicate Y (ppm) assays.

Table 11-2: Summary of performance for duplicates

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

## 11.2 Sample Preparation and Analysis

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

### 11.2.1 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.

### 11.2.2 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.

### 11.2.3 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.

### 11.2.4 Analytical Method

At ALS Chemex in Vancouver the samples are 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 (HNO<sub>3</sub>) 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 volatilized.

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.

### 11.2.5 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 given the reliance of in-situ leaching on a minimum slope angle. A Lidar survey over the property is highly recommended.

## 11.3 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 a 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 within Tantalus 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.

The data provided by Tantalus are according to industry standards and data provided is appropriate for use in the resource estimation.

## 12 Data Verification

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

SGS received the “test pits and drill holes” database (DB) in the form of 2 excel files on August 21 of 2014. Additional assays (90) were received on September 9. Some updates came up to September 11 regarding trenches. The soil investigations can be diamond drill holes, pits, percussion window sampling or trenches.

It was decided to not use the trenches (5 of them named TANT1 to TANT5 containing 200 samples) because crucial information is missing. Also the 62 auger holes named NW001 to NW053 and NW055 to NW063 were removed from the DB because grades are biased low compared to surrounding grades. Assays missing a PitID had to be removed from the DB (12). Two assays tagged "outcrop" were removed from DB. Three samples are "Listed Not Received" and were removed from DB (samples # 002280, BFPIT194-05 and I616459). Some test pits and drill holes lack elevation (442) and some missed a total depth (392). Missing depths were populated from the maximum sample depth and elevation was taken from the topography surface when needed. Some missing depths correspond to test pits and drill holes never realized (379), they were removed. Some pits (71) are labeled as: "Soil sampling starting at 0.5m depth, elevation is missing". Sounding CALD008 states: "Drillhole planned but not drilled". It was removed from our DB for the estimation. None have any assays so they were removed from our DB for the estimation. Three test pits and drill holes miss coordinates are were removed from DB (TAND014, TPIT662 and TPIT663). Some test pits and drill holes miss an azimuth and dip information, they were assumed to be vertical and downward (351). There are one overlimit for the Sm and one for the Y (both >1000 ppm). They were replaced by 1000 ppm for the estimates. There are 16 Th overlimits (>1000 ppm) and 149 Nb overlimits (>2500 ppm) (assay limit used: 1000 and 2500 ppm respectively). Grab samples were removed (121). We found 21 overlaps in the assays interval that were fixed by best judgment.

The final DB contains 4771 collars (4412 test pits and 359 drill holes), 37,212 assays and 37,212 lithologies. All drill holes are vertical (dip at -90°) except for 268 (17 dip at -45° and 251 dip at -70°). There are 19 assays elements available for all of the assay intervals. None are missing. In addition, the density (humid) is available for most assay intervals (27,671), 9,541 of them are missing.

Many of the detected issues could have been avoided or detected if the database had been imported using a relational database system. SGS recommends that such a system be implemented before the next exploration campaign.

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

## 13 Mineral Processing and Metallurgical Testing

### 13.1 Historical Testwork

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

#### 13.1.1 Soviet Mineralogical Testwork

Mineralogical testwork completed as part of the Soviet Geological Mission confirmed that the locally and historically termed fasibitkite 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 ( $\pm$  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<sub>2</sub>O<sub>5</sub>, 2.31 % Ta<sub>2</sub>O<sub>5</sub>, 1.10 % ZrO<sub>2</sub>, 0.35 % ThO<sub>2</sub> 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.

### 13.1.2 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;
- 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.

## 13.2 Pre 2013 Testwork

Contemporary mineralogical testwork and studies have been completed in Germany by independent Geochemist Dr. Udo Jakobs ([www.dr-jakobs-gmbh.de](http://www.dr-jakobs-gmbh.de)) and Consulting Geologist Dr. Thomas Hatzl ([www.mineral-consult.de](http://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.

### 13.2.1 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 13-1.

Table 13-1: Summary of the XRD results for sample TANT2-477067

<i>Fraction</i>	<i>Kaolinite-D</i>	<i>Illite</i>	<i>Quartz</i>	<i>Hematite</i>	<i>Goethite</i>	<i>Gibbsite</i>	<i>Baddeleyite</i>
<b>Total sample</b>	20	n.d.	50	3	12	15	n.d.
<b>&lt; 2 µm</b>	65	1	5	5	13	10	< 1
<b>&gt; 40 µm</b>	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 (ZrO<sub>2</sub>) 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 ytropyrochlor 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 saprolith material collected from the Caldera prospect (composite sample I679066 - I679069).

A summary of the XRD results for sample I679066 - I679069 are provided in Table 13-2.

Table 13-2: Summary of the XRD results for sample I679066 - I679069 (Gilbertson, 2013)

<b>Probe</b>	<b>Smectite</b>	<b>Mica</b>	<b>Illite- Smectite</b>	<b>Kaolinite- Smectite</b>	<b>Kaolinite- D</b>	<b>Chlorite</b>	<b>Quartz</b>	<b>Albite</b>	<b>K- feldspar</b>	<b>Hematite</b>	<b>Goethite</b>	<b>Gibbsite</b>
<b>Total sample</b>	1	11	7	18	21	3	33	n.d.	n.d.	2	3	1
<b>&lt;0.1mm A</b>	1	9	6	13	22	2	39	n.d.	n.d.	2	3	3
<b>&lt;0.1mm B</b>	1	10	4	17	22	2	34	n.d.	n.d.	2	3	4
<b>&lt;0.1mm C</b>	1	10	6	18	25	2	30	n.d.	n.d.	3	3	2
<b>0.1-0.315 mm MAG1</b>	1	9	6	11	24	2	38	n.d.	n.d.	3	3	3
<b>0.1-0.315 mm NONMAG1</b>	1	5	4	2	2	1	78	<1	1	<1	<1	3
<b>0.1-0.315 mm MID1</b>	1	9	6	13	21	2	39	n.d.	n.d.	2	3	4
<b>0.1-0.315 mm B'</b>	1	4	3	5	5	1	73	n.d.	<1	1	1	5
<b>0.1-0.5 mm</b>	1	9	6	9	15	2	49	<1	n.d.	2	2	4
<b>0.5-1.0 mm</b>	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 saprolith sample returned very different results to those obtained from the ferruginous zone sample. The saprolith 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 saprolith sample. Despite being mineralised, no REE or other rare metal bearing minerals were identified in the saprolith 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).

### 13.2.2 University of Toronto Testwork

In January 2012, the University of Toronto (UoT) in Canada initiated metallurgical testwork on samples from the Tantalus project. The testwork was more specifically completed by the Department of the Chemical Engineering and Applied Geochemistry by Dr. Georgiana Moldoveanu and Prof. Vladimiro 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 13-3.

Table 13-3: Summary of the samples provided to the University of Toronto (Moldoveanu, 2013)

<i>Tantalus SampleID</i>	<i>UoT SampleID</i>	<i>Prospect</i>	<i>Type</i>	<i>From (m)</i>	<i>To (m)</i>	<i>Interval (m)</i>	<i>Material</i>
I618258	MC1	Caldera	Pit sample	6.50	7.00	0.50	Saprolith
I618440	MC2	Caldera	Pit sample	5.50	6.00	0.50	Saprolith
L546213	MC3	Caldera	Pit sample	4.00	4.50	0.50	Saprolith
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	Saprolith

MC = Madagascar Clay

\* Due to the inadvertent modification of the Tantalus SampleID's prior to their arrival at the UoT, it has not yet been possible to reconcile the results with the original sample details.

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:HNO<sub>3</sub>, vol/vol) to bring the constituent elements into solution (except for the insoluble aluminosilicate matrix). The digested residue was then filtered, washed with 5% HNO<sub>3</sub> and denaturated 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<sub>2</sub>O);
- 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 denaturated 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<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> (i.e., 1M NH<sub>4</sub><sup>+</sup> exchange ions); 1M NaCl; ~ 0.5M NaCl (simulated seawater);
- S/L = 1/2 (wt/vol), i.e. 50 g clay /100 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 13-4.

Table 13-4: University of Toronto sample descriptions and aqua regia digestion results (Moldoveanu, 2013)

<i>UoT SampleID</i>	<i>UoT Description</i>	<i>% Dissolved during ARD</i>
<b>MC1</b>	Light brown, very fine powder	23.6
<b>MC2</b>	Pinkish-orange, soft chunks (easily broken with a pestle) plus some fine black sandy magnetic material	35.3
<b>MC3</b>	Pinkish-orange, very fine, occasional soft chunks (easily broken with the pestle)	31.5
<b>MC4</b>	Pinkish-orange, higher content of coarse particles (sand-like)	25.3
<b>MC5</b>	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 13-5 and Table 13-6 respectively.

Table 13-5: Total Rare Earth Oxide (TREO) results (as wt. %) (Moldoveanu, 2013)

<i>REO</i>	<i>MC1</i>	<i>MC2</i>	<i>MC3</i>	<i>MC4</i>	<i>MC5</i>
<b>La<sub>2</sub>O<sub>3</sub></b>	0.1103	0.0627	0.2047	0.0031	0.0339
<b>Ce<sub>2</sub>O<sub>3</sub></b>	0.0476	0.0388	0.0299	0.0629	0.0204
<b>Dy<sub>2</sub>O<sub>3</sub></b>	0.0034	0.0063	0.0066	0.0007	0.0027
<b>Er<sub>2</sub>O<sub>3</sub></b>	0.0021	0.0036	0.0027	0.0086	0.0014
<b>Eu<sub>2</sub>O<sub>3</sub></b>	0.0006	0.0010	0.0011	0.0000	0.0010
<b>Gd<sub>2</sub>O<sub>3</sub></b>	0.0088	0.0097	0.0131	0.0021	0.0048
<b>Ho<sub>2</sub>O<sub>3</sub></b>	0.0007	0.0010	0.0010	0.0019	0.0008
<b>Lu<sub>2</sub>O<sub>3</sub></b>	0.0003	0.0007	0.0003	0.0002	0.0003
<b>Nd<sub>2</sub>O<sub>3</sub></b>	0.0607	0.0375	0.1159	0.0028	0.0271
<b>Pr<sub>2</sub>O<sub>3</sub></b>	0.0181	0.0112	0.0327	0.0056	0.0077
<b>Sm<sub>2</sub>O<sub>3</sub></b>	0.0115	0.0090	0.0202	0.0009	0.0051
<b>Tb<sub>2</sub>O<sub>3</sub></b>	0.0013	0.0014	0.0019	0.0002	0.0007
<b>Tm<sub>2</sub>O<sub>3</sub></b>	0.0014	0.0004	0.0002	0.0007	0.0001
<b>Y<sub>2</sub>O<sub>3</sub></b>	0.0273	0.0489	0.0362	0.0024	0.0177
<b>Yb<sub>2</sub>O<sub>3</sub></b>	0.0013	0.0035	0.0018	0.0017	0.0013
<b>TREO</b>	0.295	0.235	0.468	0.093	0.125
<b>ThO<sub>2</sub></b>	0.0064	0.0079	0.0049	0.0335	0.0066
<b>U<sub>3</sub>O<sub>8</sub></b>	0.0128	0.0283	0.0145	0.0256	0.0244
<b>Sc<sub>2</sub>O<sub>3</sub></b>	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 13-6: Relative Rare Earth Oxide (REO) results (as wt. %) (Moldoveanu, 2013)

<b>REO</b>	<b>MC1</b>	<b>MC2</b>	<b>MC3</b>	<b>MC4</b>	<b>MC5</b>
<b>La<sub>2</sub>O<sub>3</sub></b>	37.33	26.62	43.71	3.28	27.13
<b>Ce<sub>2</sub>O<sub>3</sub></b>	16.12	16.46	6.39	67.67	16.29
<b>Dy<sub>2</sub>O<sub>3</sub></b>	1.16	2.68	1.40	0.79	2.12
<b>Er<sub>2</sub>O<sub>3</sub></b>	0.72	1.51	0.59	9.21	1.16
<b>Eu<sub>2</sub>O<sub>3</sub></b>	0.22	0.41	0.24	0.01	0.78
<b>Gd<sub>2</sub>O<sub>3</sub></b>	2.98	4.10	2.79	2.26	3.85
<b>Ho<sub>2</sub>O<sub>3</sub></b>	0.24	0.44	0.21	2.05	0.67
<b>Lu<sub>2</sub>O<sub>3</sub></b>	0.08	0.29	0.06	0.23	0.24
<b>Nd<sub>2</sub>O<sub>3</sub></b>	20.56	15.93	24.75	2.98	21.69
<b>Pr<sub>2</sub>O<sub>3</sub></b>	6.11	4.77	6.98	6.00	6.16
<b>Sm<sub>2</sub>O<sub>3</sub></b>	3.88	3.81	4.31	0.96	4.11
<b>Tb<sub>2</sub>O<sub>3</sub></b>	0.43	0.59	0.40	0.20	0.56
<b>Tm<sub>2</sub>O<sub>3</sub></b>	0.46	0.16	0.05	0.74	0.08
<b>Y<sub>2</sub>O<sub>3</sub></b>	9.24	20.75	7.74	2.61	14.12
<b>Yb<sub>2</sub>O<sub>3</sub></b>	0.44	1.47	0.38	1.78	1.01
<b>TREO</b>	100	100	100	100	100

From Table 13-5 and Table 13-6 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 (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> (1M total exchange cations):**

The results of leaching with 0.5 M (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> (1M total exchange cations) are provided in Table 13-7.

**Table 13-7: REE extraction levels (as % Extraction) both as individual REE and Total REE, respectively, based on solids analysis (0.5M (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub>, 60 min, 22°C, S/L = 1/2, pH ~ 5.4) (Moldoveanu, 2013)**

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

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

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

General comments relating to the leaching with 0.5 M (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> (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 Tantalus 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 13-8.

Table 13-8: % REE Extraction during leaching with 1M NaCl (Moldoveanu, 2013)

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

Despite the fact that both 0.5M (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> 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<sup>+</sup>, 19.4 g/L Cl<sup>-</sup>, 2.7 g/L SO<sup>42-</sup>, 1.28 g/L Mg<sup>2+</sup>, 0.4 g/L K<sup>+</sup>, 0.4 g/L Ca<sup>2+</sup>) are provided in Table 13-9.

**Table 13-9: % REE Extraction during leaching with simulated seawater (0.48M Na) (Moldoveanu, 2013)**

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

### **13.2.3 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<sub>2</sub>O 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<sub>4</sub>)<sub>2</sub>SO<sub>4</sub>, 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.

$$\%E_{\text{total}} = \frac{\text{Mass REE}_{\text{leached total}}}{\text{Mass REE}_{\text{in clay initially}}} \times 100$$

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

Table 13-10: Two-stage leaching for MC3 (22°C, 60 min, S/L = 1/2) (Moldoveanu, 2013)

<i>Element</i>	<i>0.5M (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub></i>		<i>1M NaCl</i>		<i>SSW</i>	
	<i>E<sub>1</sub></i>	<i>E<sub>tot</sub></i>	<i>E<sub>1</sub></i>	<i>E<sub>tot</sub></i>	<i>E<sub>1</sub></i>	<i>E<sub>tot</sub></i>
<b>La</b>	83.6	97.5	48.8	72.0	42.8	56.9
<b>Ce</b>	0.0	0.0	0.0	0.0	0.0	0.0
<b>Dy</b>	80.9	94.7	48.6	70.2	42.7	59.4
<b>Er</b>	86.8	93.0	47.6	66.1	41.6	57.7
<b>Eu</b>	62.1	77.6	48.0	67.0	41.1	51.4
<b>Gd</b>	82.6	96.7	48.1	70.4	44.1	59.4
<b>Ho</b>	75.5	93.2	57.7	80.0	48.2	63.0
<b>Lu</b>	52.3	61.2	12.2	12.2	5.0	5.0
<b>Nd</b>	80.8	94.3	44.4	65.5	40.1	52.6
<b>Pr</b>	75.1	87.1	41.3	60.7	36.6	48.0
<b>Sm</b>	90.6	94.3	50.6	74.5	46.2	61.4
<b>Tb</b>	84.1	98.5	46.4	66.4	42.6	56.1
<b>Tm</b>	53.4	60.9	61.1	77.4	49.3	57.2
<b>Y</b>	77.3	90.5	48.0	69.2	42.8	60.5
<b>Yb</b>	73.2	85.7	41.5	57.2	34.4	46.6
<b>Total REE</b>	76.6	88.8	44.0	64.6	39.1	52.1

As observed in Table 13-10, 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<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> offers the best extraction levels (Table 13-7), between 60 and 76 %, whereas 1M NaCl and simulated seawater (0.48M Na) achieve ~20 % units lower extraction levels (Table 13-8 and Table 13-9, 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 13-10).

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,  $(\text{NH}_4)_2\text{SO}_4$  and a mixture of seawater and  $(\text{NH}_4)_2\text{SO}_4$ ;
- 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.

### 13.3 Post 2014 Testwork

#### 13.3.1 Outotec Testwork

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

##### 13.3.1.1 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
  - $(\text{NH}_4)_2\text{SO}_4$ , 0.50 mol/l and 0.25 mol/l
  - NaCl, 0.48 mol/l (corresponds concentrations in sea water)
  - $\text{Na}_2\text{SO}_4$ , 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  $(\text{NH}_4)_2\text{SO}_4$  solution, 0.25 mol/l  $(\text{NH}_4)_2\text{SO}_4$  solution, 0.48 mol/l NaCl solution (synthetic seawater).** For all tests the Feed material was a single sample from one deposit and they was assayed for main metal analyses along with U, Th, Sc.

Leaching tests were also completed using a 0.5 mol/l (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> 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.***

#### 13.3.1.2 *Precipitation*

- Preliminary precipitation test was carried out by using oxalic acid
- Precipitate was obtained
- Na<sub>2</sub>CO<sub>3</sub> 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.

#### 13.3.1.3 *Process concept development and modelling*

- Process concept by using (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> leaching and oxalic acid precipitation as a starting case

#### Main process steps

- 3 Precipitation of REE
- 1 Leaching
- 2 Solid-liquid separation
- 4 Solid-liquid separation of REE cake
- 5 Oxalate and gypsum removal
- 6 Solution recycling
- 7 Bleed stream out
- 8 Solid residue out
- 9 REE product (RE oxide)

#### 13.3.1.4 *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<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> 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  $\text{Na}_2\text{CO}_3$
- The main metals seem to remain in the solution during the precipitation (which is important feature: possibilities for purer product)

### 13.3.2 SGS Lakefield Testwork

#### 13.3.2.1 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 TRE 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 13-11: Sample Inventory

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

**Table 13-11: Sample Inventory**

SGS Sample ID	Tantalus Sample ID	Sample Type	Pit ID	Depth		No. of bags	Weight kg
				from	to		
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

Table 13-12: Sample Inventory (Cont.)

SGS Sample ID	Tantalus Sample ID	Sample Type	Pit ID	Depth		No. of bags	Weight kg
				from	to		
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

### 13.3.2.2 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 homogenized 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 13-13 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 13-13: 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

### 13.3.3 SGS Lakefield Hydrometallurgical Testing

#### 13.3.3.1 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.

### 13.3.3.2 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 summarized in Table 4. 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 deionized (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 13-14. The residue and solution assay data are summarized in Table 13-16 and Table 13-17, 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 13-14: 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	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	NaCl	NaCl	NaCl	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	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 13-15: Contact Methodology – REE Extraction (extraction values are 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 from 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,  $(\text{NH}_4)_2\text{SO}_4$ ) and CM8 (4 contacts + no wash,  $(\text{NH}_4)_2\text{SO}_4$ ). 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 differences 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 13-16: 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 13-17: 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

**13.3.3.3 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 13-18. Table 13-19 shows the extractions from the Optimum Eluant tests. The residue and solution assay data are summarized in Table 13-20 and Table 13-21, 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 13-18: 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 13-19: 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 13-20: 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 13-21: 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 13-1 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 13-1 for comparison reason (these tests were performed using the “1 contact plus 3 displacement washes” methodology, too).

As shown in Figure 13-1 (right) REE extractions were very similar in all the tests where 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.

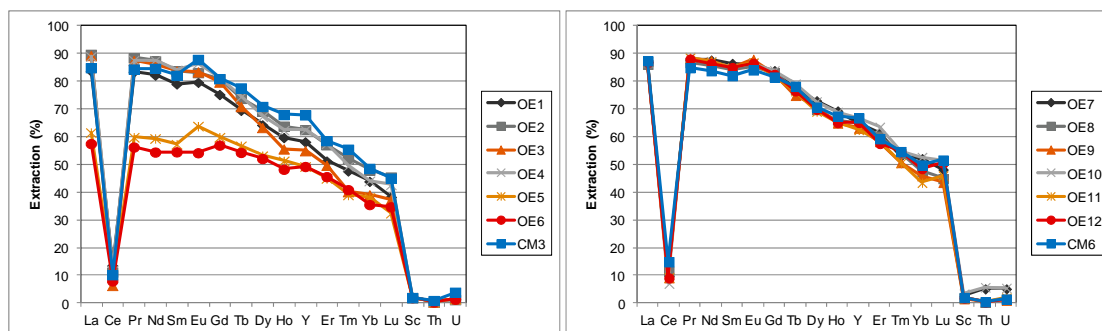


Figure 13-1: Optimum Eluant REE Extractions

Tests where sodium chloride was used as eluant (Figure 13-1, 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.

#### 13.3.3.4 SGS Lakefield Isotherms Tests

A total of 4 tests were performed in order to built 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 summarized in Table 13-22. The extractions plus residue and solution assay data are presented in Table 13-23.

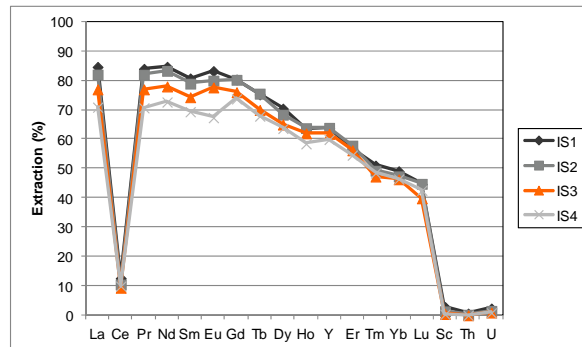
**Table 13-22: Isotherms - Test Conditions**

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 <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>
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 13-23: Isotherms - Extractions, Residue and Solution Assays**

Parameter Element	Extractions (%)				Residue assays (% g/t)				Solution assays (mg/L)			
	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 13-2 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 13-2: Isotherms REE Extractions**

Figure 13-3 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.

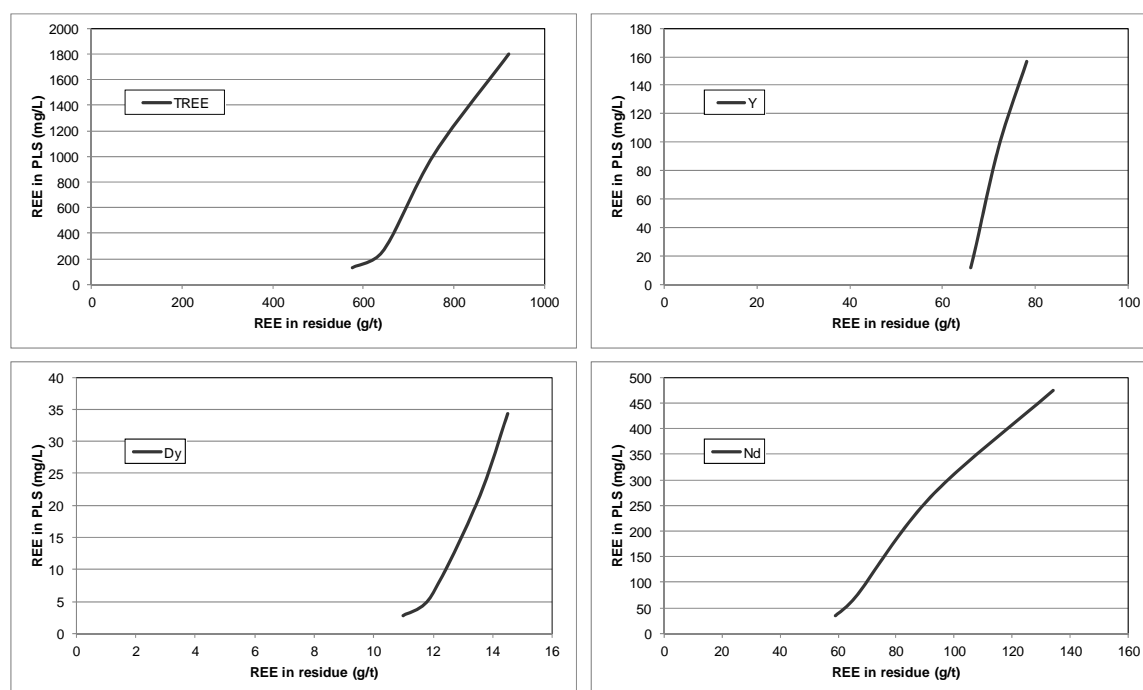


Figure 13-3: TREE, Y, Nd and Dy Isotherms

### 13.3.4 SGS Lakefield Heap Leaching Tests

#### 13.3.4.1 SGS Lakefield Tests Design and Feed Preparation

After all three series of shaking tests were finalized 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<sup>2</sup> (equivalent to 0.5 mL/min);
- Room temperature;
- Running time of 218 hours;
- DI water washing at 60 L/h/m<sup>2</sup> 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 13-4. 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 13-4: Column Tests Agglomerated Feed - Column 1 (left) and Column 2 (right)

#### 13.3.4.2 SGS Lakefield Column Operation

Figure 13-5 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 13-5: Photograph of Column Set-up

Table 13-22 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<sup>2</sup> and 12.8 L/h/m<sup>2</sup> 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<sup>2</sup> and 12.8 L/h/m<sup>2</sup> 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<sup>2</sup>, 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 13-24: Heap Leaching - Main Parameters**

Parameter	Unit	CL1	CL2
Feed Sample	-	Clay composite	Clay composite
Reagent	-	(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	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 <sup>2</sup>	14.1	12.8
Avg. Discharge Rate	L/h/m <sup>2</sup>	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

\*This is the time that took to start discharging.

#### 13.3.4.3 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 13-25 and Table 13-26 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 13-25: Column 1 - Extractions (values reported as %)**

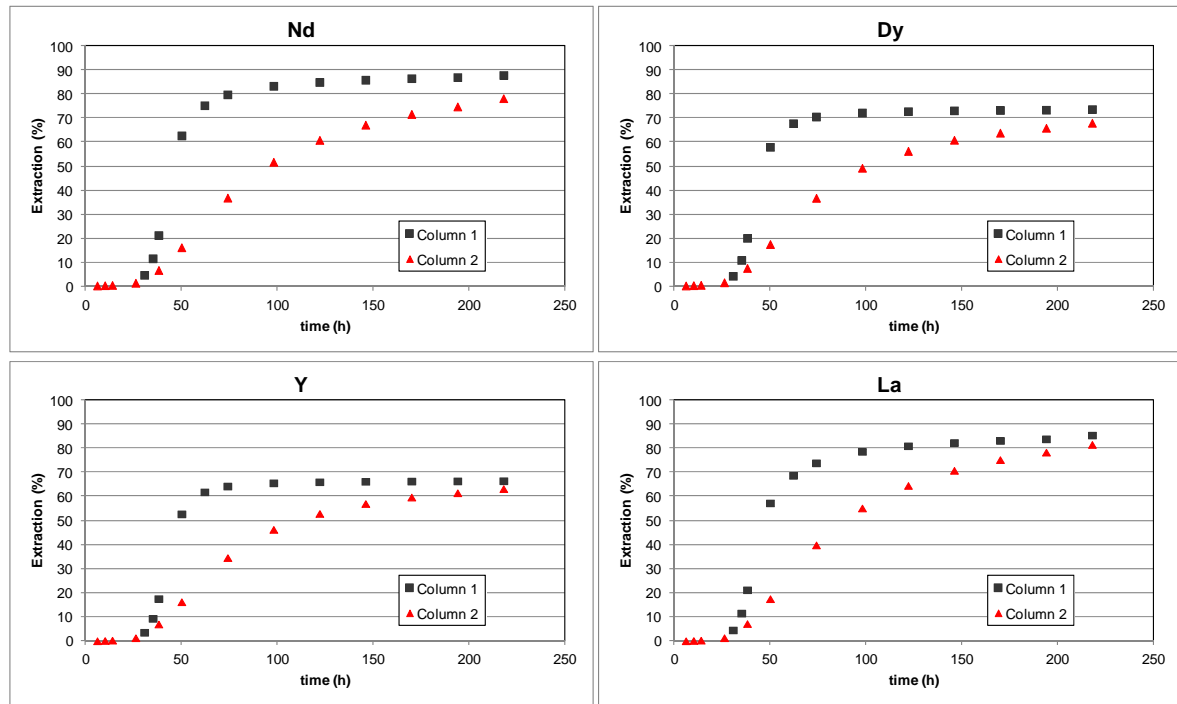
Element	31 h	35 h	38 h	50 h	62 h	74 h	98 h	122 h	146 h	170 h	194 h	218 h
Si	0	0	0	0	0	0	0	0	0	0	0	0
Al	0	0	0	0	0	0	0	0	0	0	0	0
Fe	0	0	0	0	0	0	0	0	0	0	0	0
Mg	0	1	1	3	4	4	4	4	4	4	4	4
Ca	1	2	4	10	12	14	15	15	15	16	16	16
P	0	0	0	1	1	1	2	3	3	4	5	8
Mn	0	0	0	1	2	2	2	2	2	2	2	2
La	5	12	21	57	69	74	79	81	83	83	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 13-26: 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 13-6. 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 13-6: 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 13-7 along with OE8 and OE2 extractions (from the SGS Lakefield Optimum Eluant Tests).

When comparing the tests using ammonium sulphate as eluant (Figure 13-7, 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.

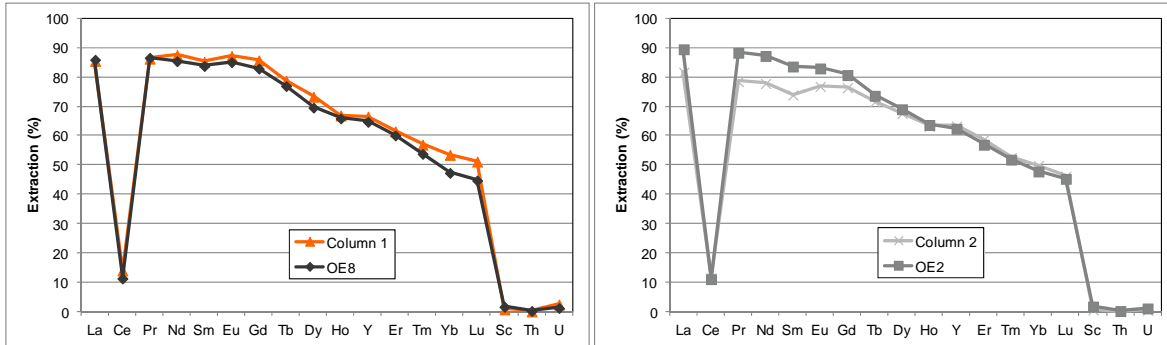


Figure 13-7: Heap Leaching Extractions vs. Optimum Eluant Extractions

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 13-8, 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 13-27 and Table 13-28, respectively.

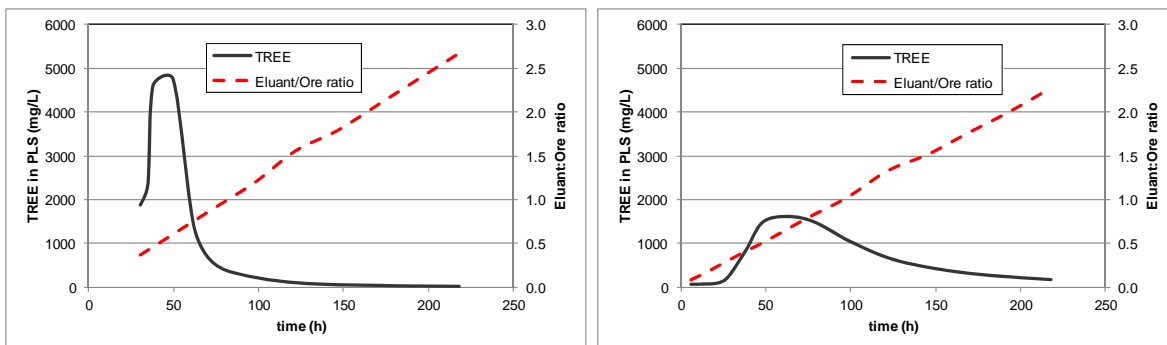


Figure 13-8: Heap Leaching - Column 1 (left) and Column 2 (right) Discharge TREE tenors

Table 13-27: 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 13-28: 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

### 13.3.5 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 19.

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 13-29: 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

### 13.3.5.1 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 13-30 shows the main test conditions. The precipitation efficiencies for these tests are presented in Table 21.

**Table 13-30: 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 <sub>2</sub> C <sub>2</sub> O <sub>4</sub> Conc.	10%	10%	10%	10%	10%
stoich H <sub>2</sub> C <sub>2</sub> O <sub>4</sub> :REE	72%	145%	217%	250%	300%

**Table 13-31: 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.

### 13.3.5.2 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 13-32 shows the main test conditions. The precipitation efficiencies for these tests are presented in Table 23.

**Table 13-32: 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 <sub>4</sub> HCO <sub>3</sub> Conc.	50 g/L	50 g/L	50 g/L
pH target	5.0	5.5	6.5

**Table 13-33: 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 13-9 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.

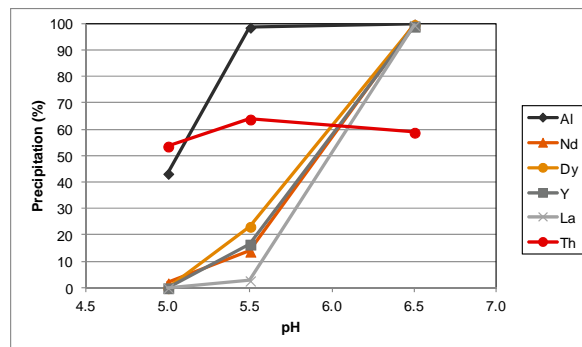


Figure 13-9: Aluminium Removal - Precipitation vs. pH

### 13.3.6 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 ratio 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<sup>2</sup> 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.

### 13.4 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.

More hydrological work will be required to evaluate if in-situ leaching is going to be considered as the primary extraction method. These tests are difficult to undertake in a lab scenario due to impossibility

of preserving in-situ conditions during extraction and transport of material. This suggests undertaking pilot tests on the site itself with all the logistical and permitting implications

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 programme will provide valuable information as to the variability in the proportion of REEs in the deposit that are readily extractable, both across the lateral extent of the orebody, with depth, with the variable parent rock and grade of the material itself.

## 14 Mineral Resource Estimates

### 14.1 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 meters. 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 mineralized 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

### 14.2 Validation of the Database

SGS received the final “test pits and drill holes” database (DB) in the form of 2 excel files on August 21 of 2014. Additional assays (90) were received on September 9. Some updates came up to September 11 regarding trenches. Please see section 12 for more details on the verification and changes that were made to the database prior to resource estimation.

Table 14-1: Variables in the DB and Used in the Resource Estimation

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

Table 14-2: Calculated Variables

Number	Calculated Variable	Units	Formula	Number	Calculated Variable	Units	Formula
1	Y <sub>2</sub> O <sub>3</sub>	ppm	Y/0.7874	14	Yb <sub>2</sub> O <sub>3</sub>	ppm	Yb/0.8782
2	La <sub>2</sub> O <sub>3</sub>	ppm	La/0.8527	15	Lu <sub>2</sub> O <sub>3</sub>	ppm	Lu/0.8794
3	Ce <sub>2</sub> O <sub>3</sub>	ppm	Ce/0.8538	16	Nb <sub>2</sub> O <sub>5</sub>	ppm	Nb/0.6990
4	Pr <sub>2</sub> O <sub>3</sub>	ppm	Pr/0.8545	17	Ta <sub>2</sub> O <sub>5</sub>	ppm	Ta/0.8190
5	Nd <sub>2</sub> O <sub>3</sub>	ppm	Nd/0.8574	18	ThO <sub>2</sub>	ppm	Th/0.8788
6	Sm <sub>2</sub> O <sub>3</sub>	ppm	Sm/0.8624	19	U <sub>3</sub> O <sub>8</sub>	ppm	U/0.8480
7	Eu <sub>2</sub> O <sub>3</sub>	ppm	Eu/0.8636	20	TREO	ppm	LREO+HREO
8	Gd <sub>2</sub> O <sub>3</sub>	ppm	Gd/0.8676	21	TREOnoCe	ppm	TREO-Ce <sub>2</sub> O <sub>3</sub>
9	Tb <sub>2</sub> O <sub>3</sub>	ppm	Tb/0.8688	22	CREO	ppm	Nd <sub>2</sub> O <sub>3</sub> +Y <sub>2</sub> O <sub>3</sub> +Eu <sub>2</sub> O <sub>3</sub> +Tb <sub>2</sub> O <sub>3</sub> +Dy <sub>2</sub> O <sub>3</sub>
10	Dy <sub>2</sub> O <sub>3</sub>	ppm	Dy/0.8713	23	HREO	ppm	Y <sub>2</sub> O <sub>3</sub> +Eu <sub>2</sub> O <sub>3</sub> +Gd <sub>2</sub> O <sub>3</sub> +Tb <sub>2</sub> O <sub>3</sub> +Dy <sub>2</sub> O <sub>3</sub> +Ho <sub>2</sub> O <sub>3</sub> +Er <sub>2</sub> O <sub>3</sub> +Tm <sub>2</sub> O <sub>3</sub> +Yb <sub>2</sub> O <sub>3</sub> +Lu <sub>2</sub> O <sub>3</sub>
11	Ho <sub>2</sub> O <sub>3</sub>	ppm	Ho/0.873	24	LREO	ppm	La <sub>2</sub> O <sub>3</sub> +Ce <sub>2</sub> O <sub>3</sub> +Pr <sub>2</sub> O <sub>3</sub> +Nd <sub>2</sub> O <sub>3</sub> +Sm <sub>2</sub> O <sub>3</sub>
12	Er <sub>2</sub> O <sub>3</sub>	ppm	Er/0.8745				
13	Tm <sub>2</sub> O <sub>3</sub>	ppm	Tm/0.8756				

### 14.3 Selection of the Mineralized Intervals

The selection of mineralized intervals (MI) in the test pits and drill holes was executed solely by the lithologies. The first layer encountered from the surface is usually summarized 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 SST 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” mineralized interval was selected randomly and one of the “SAP” mineralized interval was selected randomly. In total, 21 “PED” and 19 “SAP” mineralized intervals were removed from the estimation.

The count and lengths of the final mineralized intervals, along with the contained assay statistics, and the contained gaps (lengths not assayed) are summarized in the Table 14-3. The Figure 14-1 shows some test pits and drill holes on section centered at 8,469,900 mN with actual proportions. The Figure 14-2 shows the same section with compressed horizontal distances (equivalent to a vertical exaggeration of 10).

Table 14-3: Count and Lengths of Mineralized Intervals, Contained Assays, and Gaps

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



Figure 14-1: Normal View of Test pits and drill holes with Mineralized Intervals (MIs)

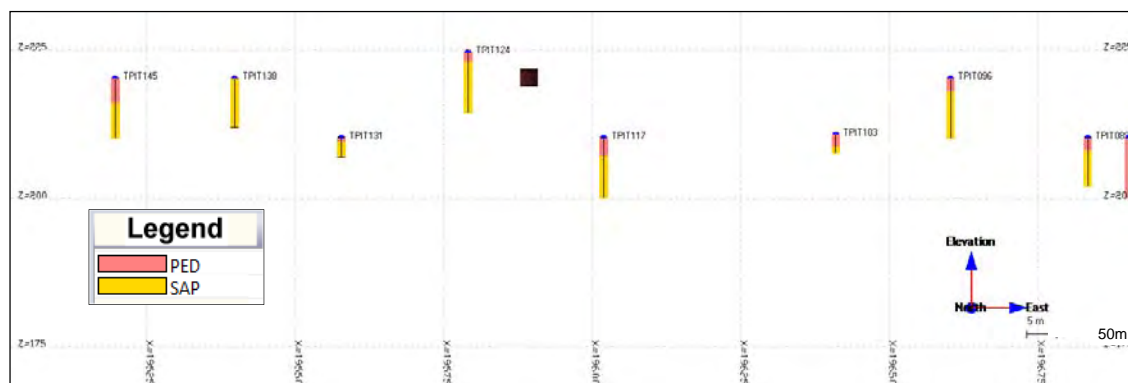


Figure 14-2: View of Test pits and drill holes with Mineralized Intervals shown with a 10x vertical exaggeration

## 14.4 Validation of the Density

On October 21, we received 4569 dry densities measurements. We validated the measurements by plotting the dry density vs the size of the samples, and the dry density vs the wet density. In the end, we kept 4,309 dry densities measurements and 4,126 of them that have an appropriate wet density enabling to estimate the water content (mass of water per unit mass of dry specimen). The Table 14-4 shows statistics on retained density measurements and Table 14-5 shows the dry densities chosen for the resource estimation. More statistics were done to verify if the density of the “PED” and “SAP” layers were distinguishable and also to verify if the density was related to the grade. Nothing conclusive was found. There is variability in the density and the water content but only the geographic location seems to have a clear relationship with it.

**Table 14-4: 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%
<b>TOTAL</b>	<b>21,113</b>	<b>1.40</b>	<b>1.53</b>	<b>1.66</b>	<b>1.53</b>	<b>4,309</b>	<b>1.01</b>	<b>1.11</b>	<b>1.21</b>	<b>1.12</b>	<b>4,126</b>	<b>26%</b>	<b>35%</b>	<b>45%</b>	<b>36%</b>

**Table 14-5: Densities Used for the Resource Estimation**

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

## 14.5 Selection of the Topography Surface to be Used for the Model

SGS originally supplied with 4 available topography surfaces and with a 5<sup>th</sup> on September 9<sup>th</sup>:

- Collars elevation data: covers all the project area, was taken by consumer grade handheld GPS. High accuracy but too low precision for use
- Government map topography (contours): covers more than the project area. Contours every 25m. Medium accuracy (after SGS custom corrections) and medium-low precision
- 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
- Fugro geophysical survey elevation information: covers 17% of the project area. Medium-high accuracy (after SGS custom corrections) and medium-high precision

- Digitized topo map (10m contours apparently from Government maps and Fugro combined): it covers exactly the project area. Contours every 10 m. Medium-high accuracy (no corrections needed) and medium precision.

It was decided to use this last topography surface for the resource model. The collar elevation information is not usable because while it is quite accurate, it has very low precision (local errors of up to 80m). All sounding elevations were replaced by the elevation of the topography surface before the modeling.

### 14.6 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”.

### 14.7 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 14-3) 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 14-6. A perspective view of the BM is visible in Figure 14-4. The Figure 14-5 shows areas with from 1 to 6 blocks thick (equivalent to 1 to 6 meters).

Table 14-6: 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

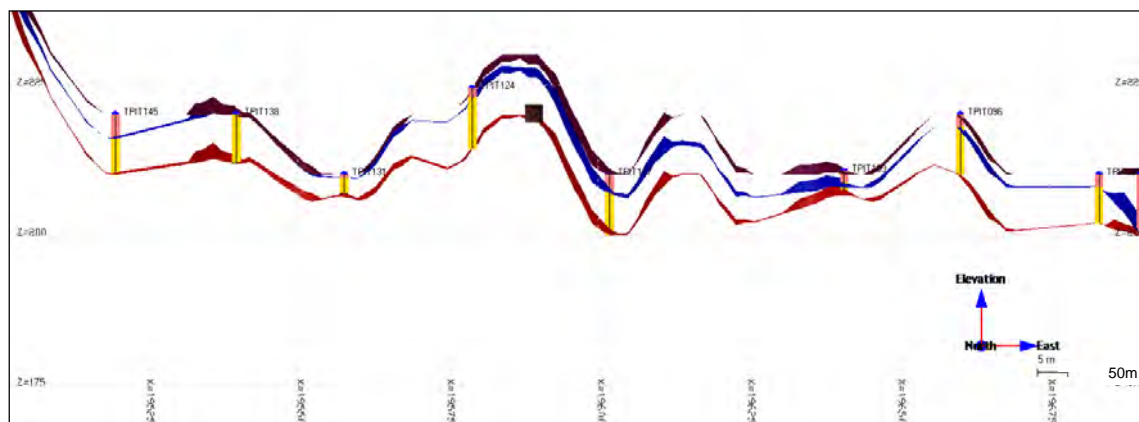


Figure 14-3: View of Test pits and drill holes with Mineralized Intervals (MIs) with a 10x vertical exaggeration

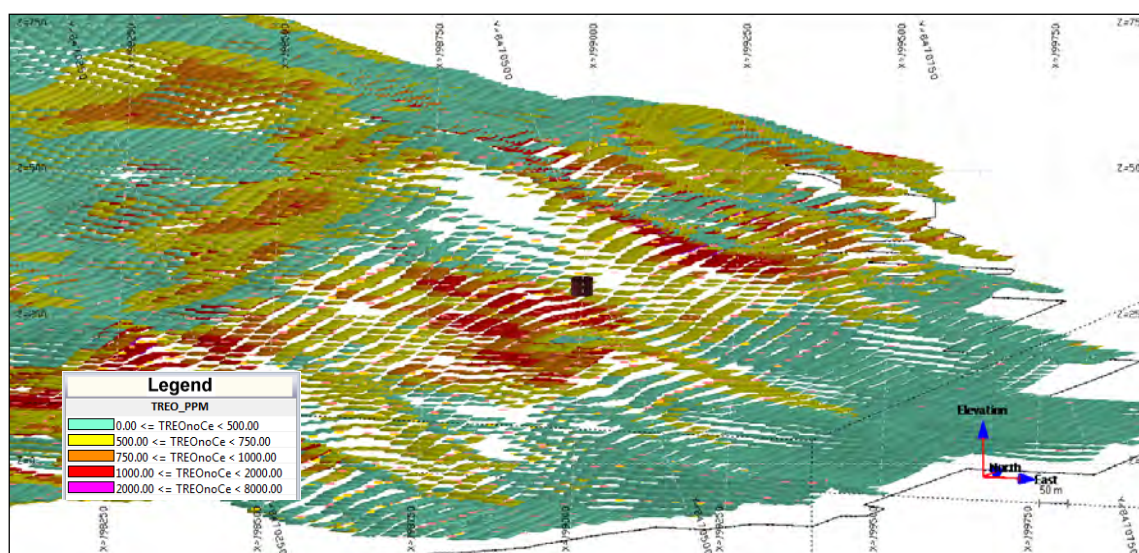


Figure 14-4: Perspective View on the BM with Colors in Function of TREOnoCe

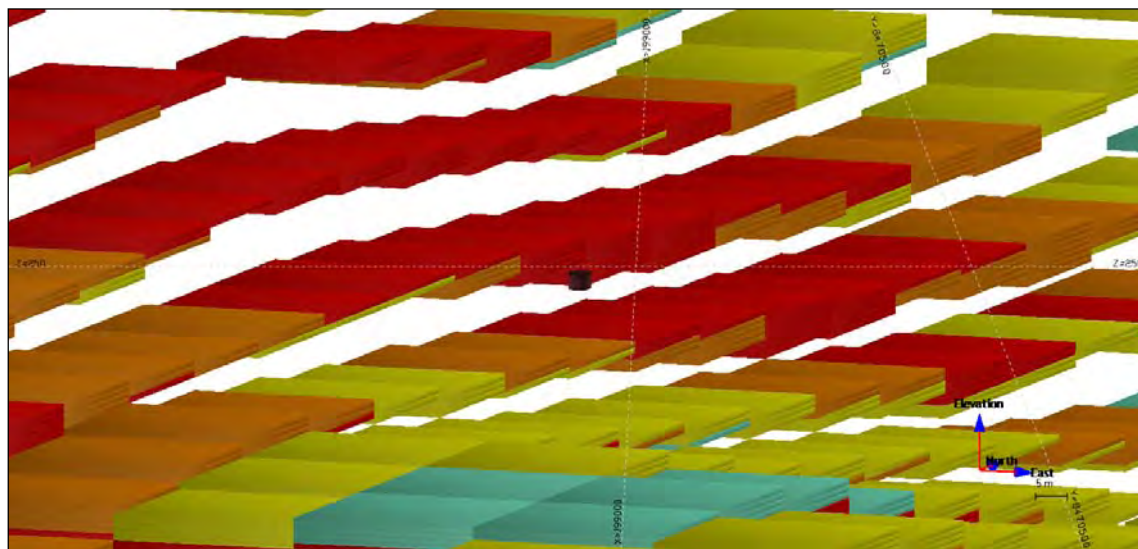


Figure 14-5: Perspective View on the BM with Colors in Function of TREOnoCe (Zoom)

## 14.8 Estimation of the Grades in the 3D Block Model

It was decided to use the ordinary kriging to estimate the 3D block model (BM). This method applies well to this distribution of mineralization. The steps required for the estimation are:

- Verification of the capping required on the assays
- Modeling of variograms for the 19 variables (see Table 14-1) for each of the 2 layers
- Estimation of the BM for the 19 variables by ordinary kriging
- Calculation of the 25 calculated variables for each block (see Table 14-2)

### 14.8.1 Capping of the Assays

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, we removed assay intervals shorter than 0.5 m for the following capping study exercise. We looked at shorter assays separately to make sure our conclusions on the requirement of a capping was still sound.

We looked at the “metal contribution” of the highest 1% of the assays intervals. For example, the “metal contribution” of the 292 highest assays (out of 29,236) in Nd (Neodymium) is 9%. Also both the histogram and the cumulative frequency graph (shown in Figure 14-6) on log scales do not look like the 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.

If a reasonable capping had been chosen, it would have had an impact to reduce the average grade of the deposit by less than 0.5% for all 19 of the estimated elements. It is not significant.

For all these reasons, it was decided not to use any capping. While the populations are clearly log-normal distributions, there are no “nugget” values that have a significant impact on the metal content. No capping was applied.

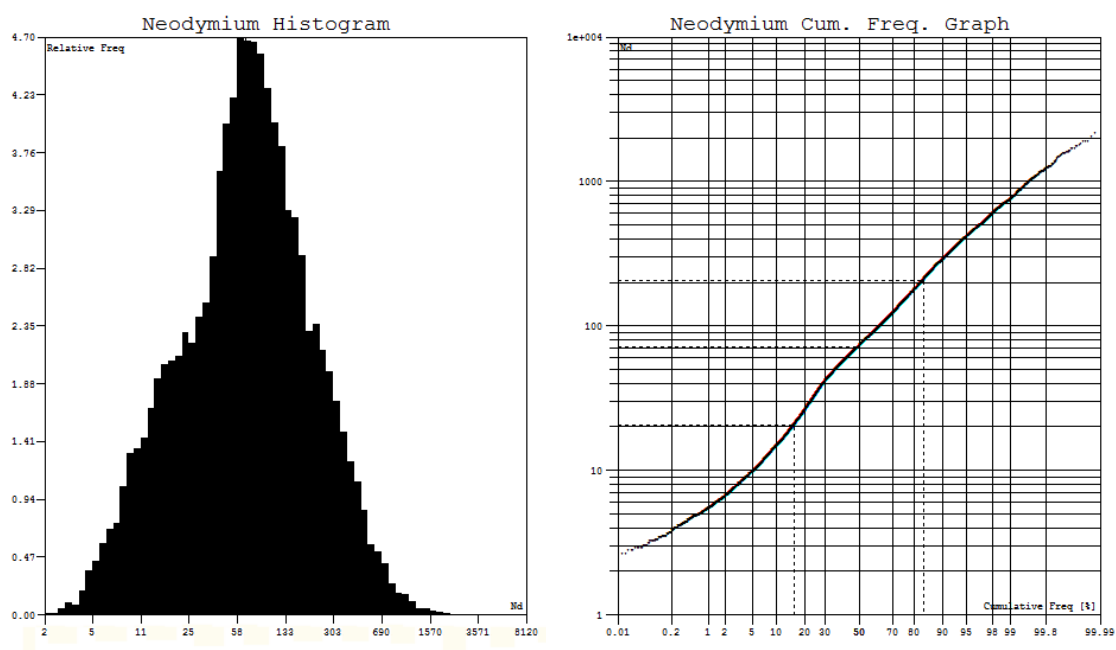


Figure 14-6: Histogram and Cumulative Frequency (Nd Assays – “PED” and “SAP” Layers)

### 14.8.2 Modeling of Variograms

In order to properly estimate the 19 variables by kriging inside the “PED” and “SAP” layers, different variogram options were considered. Because “PED” and “SAP” variograms were always different, it was decided to prepare 19 variograms for each layer. During the process, the 38 variograms were replaced by 11 meaningful variograms because many of them are considered very similar. The variograms were done in 2D using composites averaging the total thickness of the layer. Also the elevations were put 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 14-7. Variograms for Nd, Pr, Dy and Eu for PED and Nd, Pr, La and Eu for SAP are presented in Figure 14-7 to Figure 14-14 as examples of the work accomplished.

Table 14-7: 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	

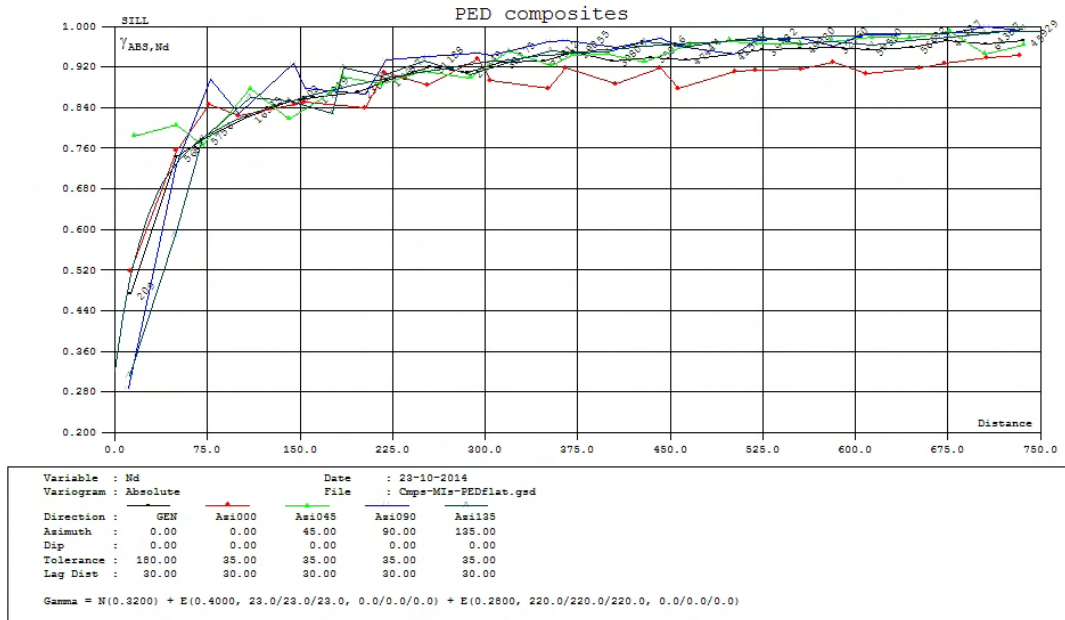


Figure 14-7: Variogram for Nd in the PED Layer

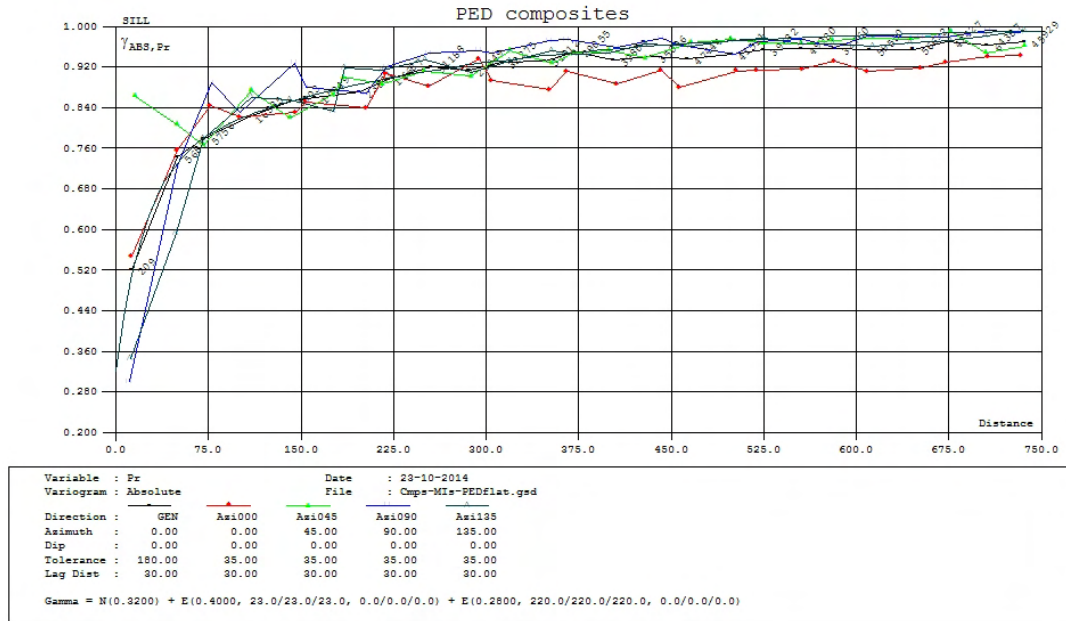


Figure 14-8: Variogram for Pr in the PED Layer

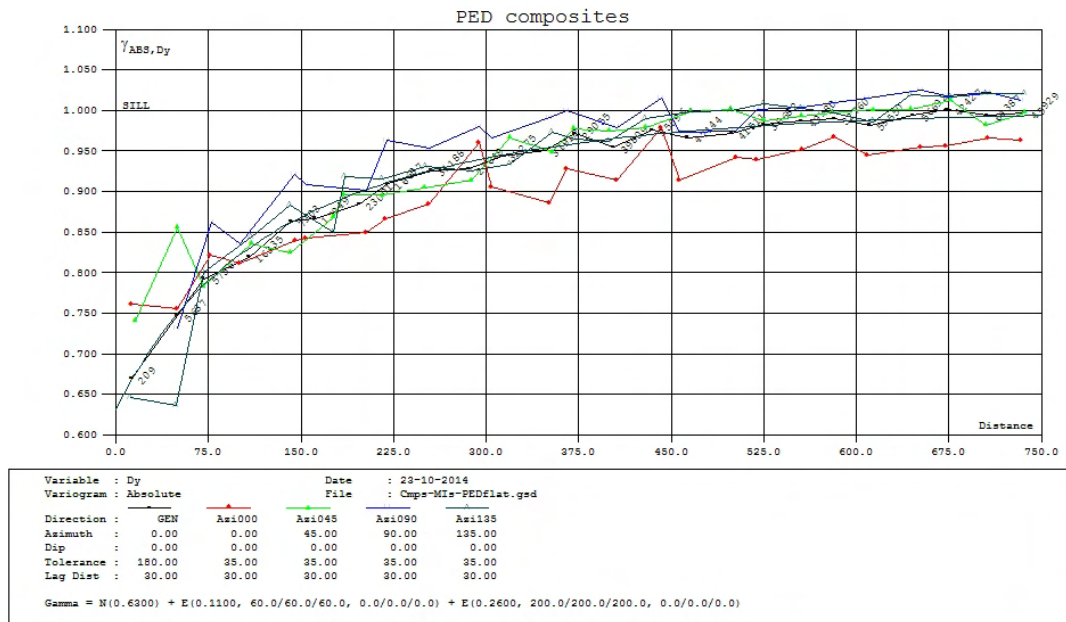


Figure 14-9: Variogram for Dy in the PED Layer

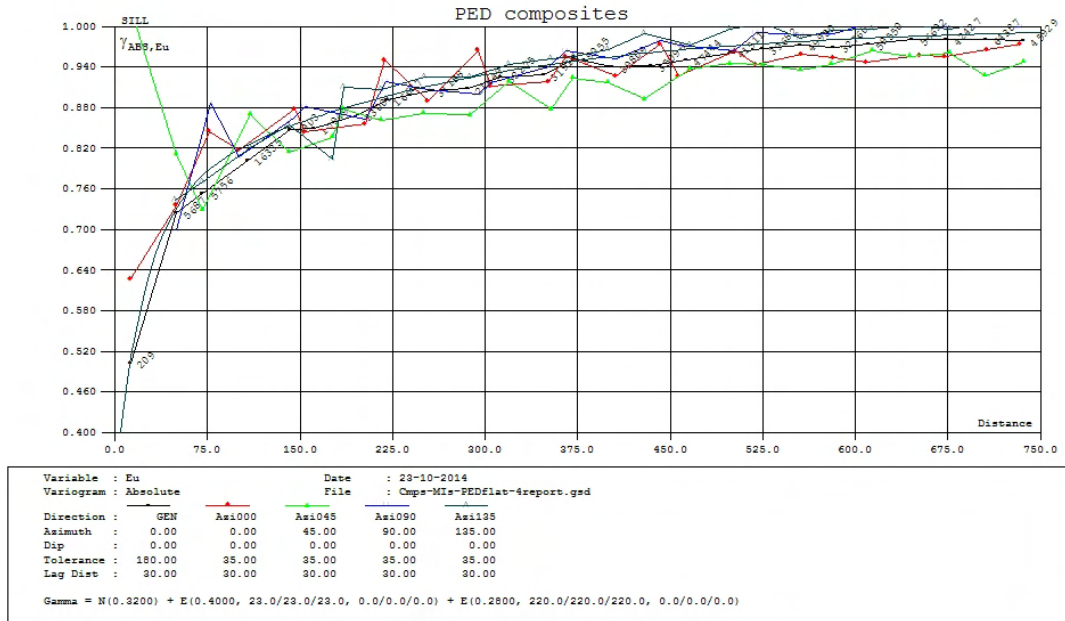


Figure 14-10: Variogram for Eu in the PED Layer

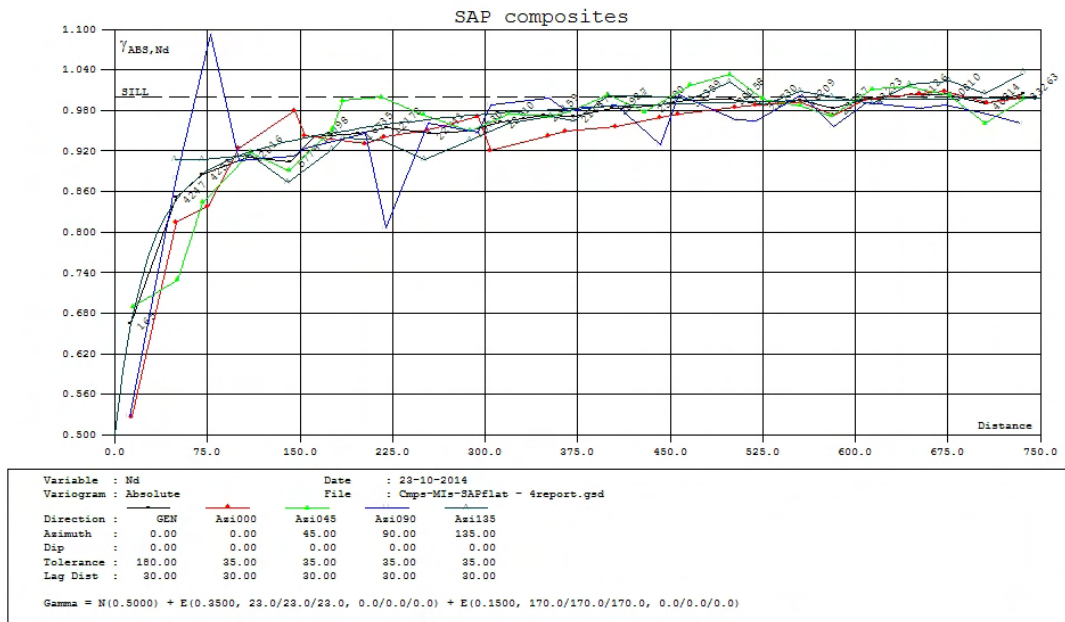


Figure 14-11: Variogram for Nd in the SAP Layer

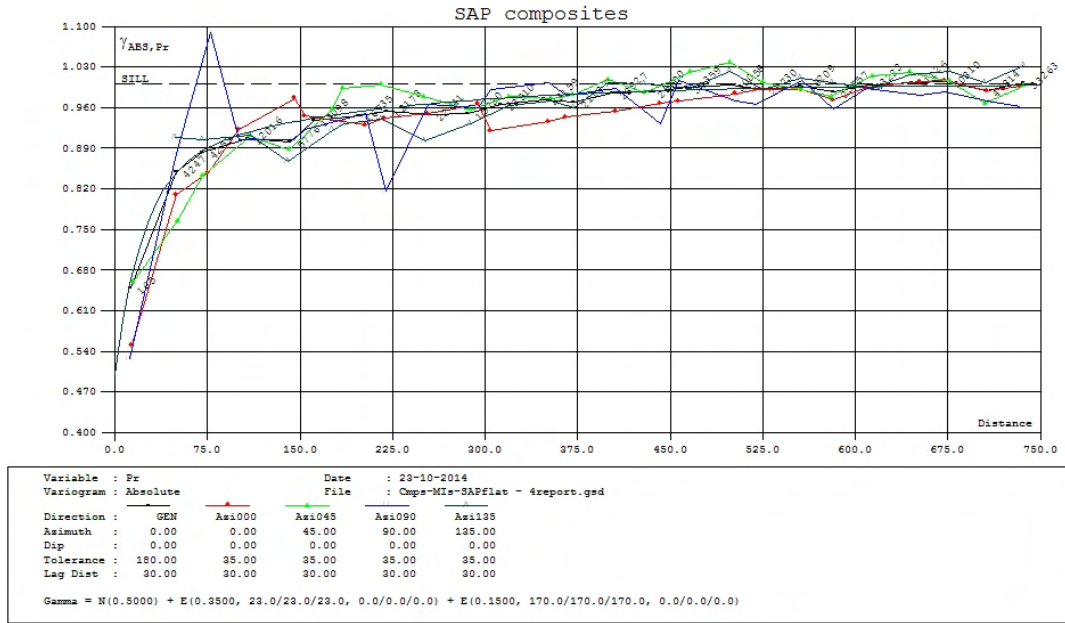


Figure 14-12: Variogram for Pr in the SAP Layer

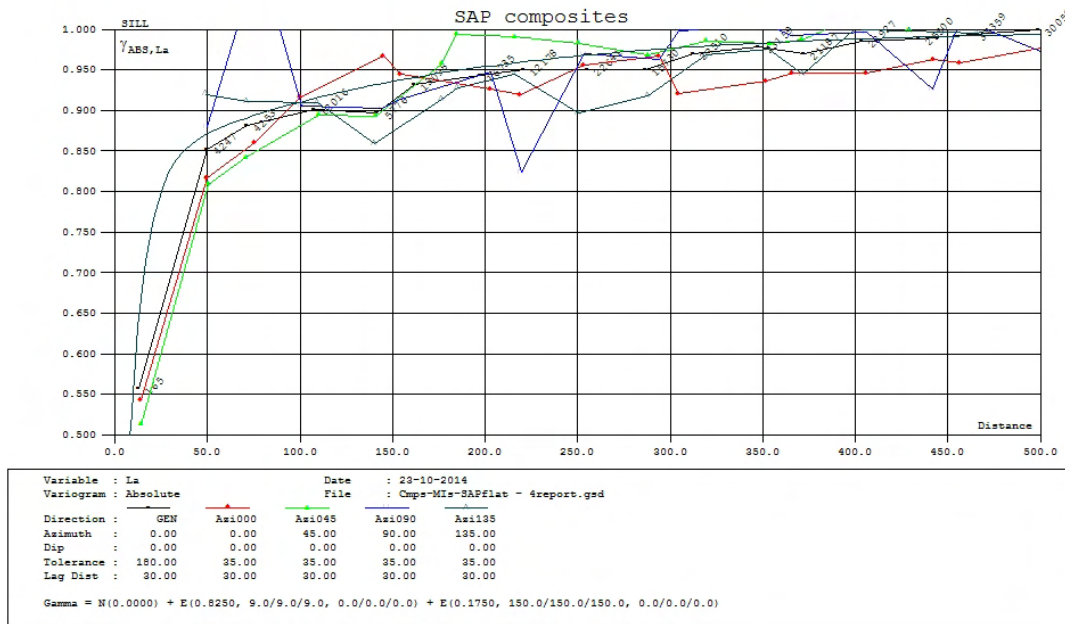


Figure 14-13: Variogram for La in the SAP Layer

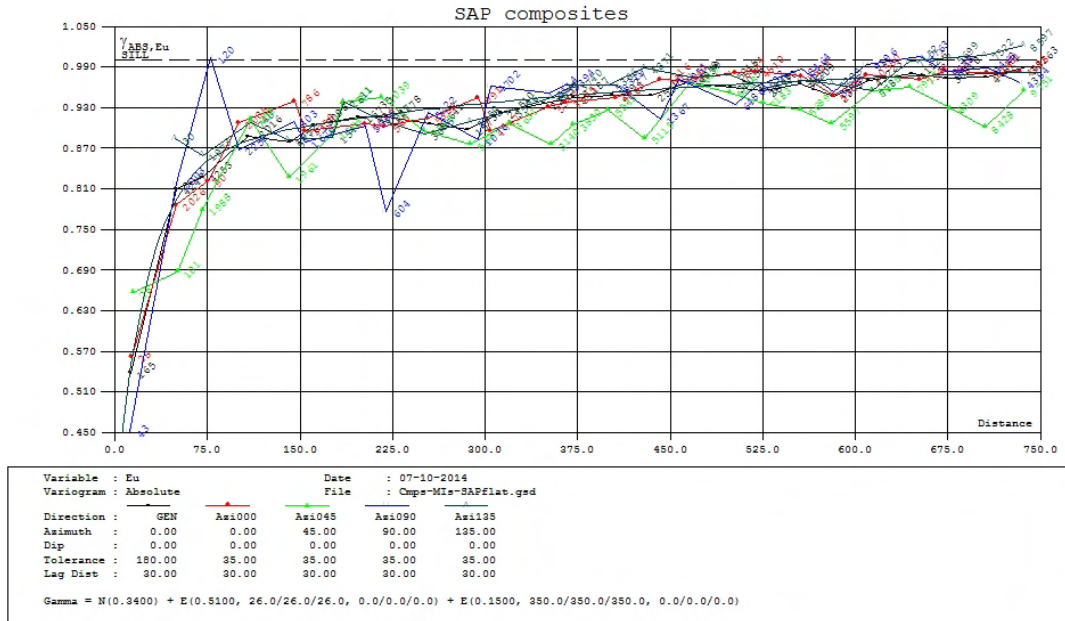


Figure 14-14: Variogram for Eu in the SAP Layer

### 14.8.3 Estimation by Kriging

The kriging was done in 3 dimensions and in 3 passes. 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 sounding” was set. Also no octant search was used. With these settings, 2% of the blocks were not estimated because less than 3 test pits and drill holes were available in a radius of 900 m.

## 14.9 Classification

### 14.9.1 Definitions

Definitions are from the Canadian institute of Mining, Metallurgy and Petroleum (CIM):

#### **Mineral Resource**

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 diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

#### **Inferred Mineral Resource**

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

### **Indicated Mineral Resource**

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

### **Measured Mineral Resource**

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

#### **14.9.2 Classification Method**

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. The Figure 14-15 shows the outlines for the different classifications of material along with the collars of the test pits and drill holes.

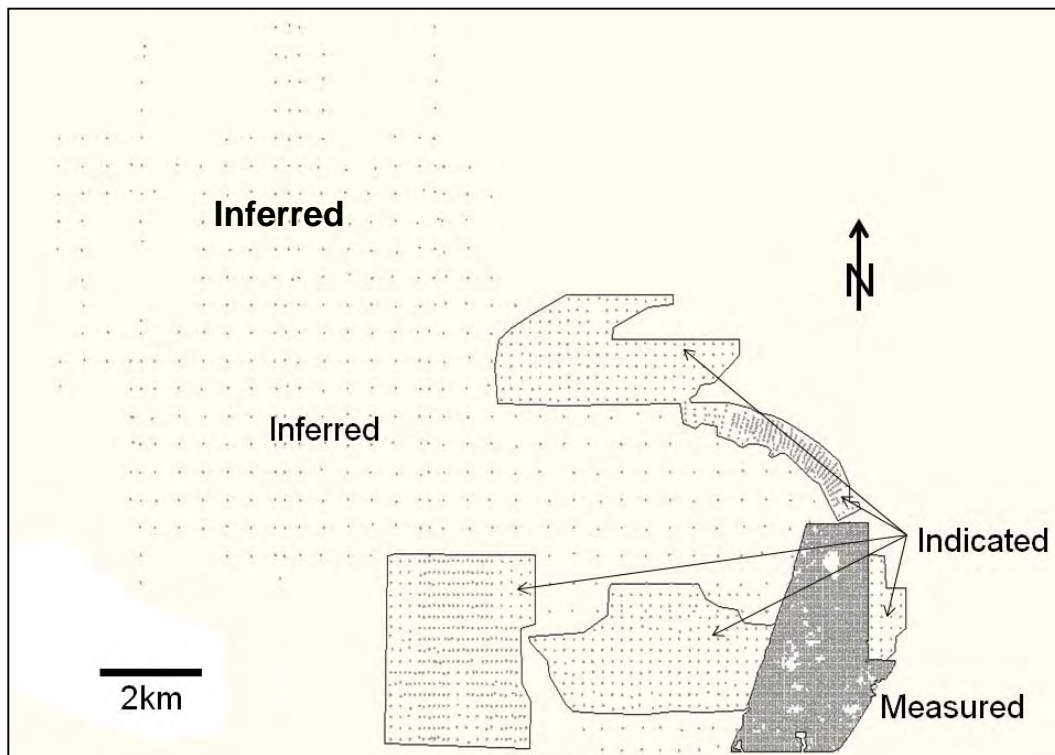


Figure 14-15: Classification of the Resource Using Test pits and drill holes Grid

#### 14.10 Inclusion of Permit Outlines to Constrain the Resource

Some of the test pits in the North West Territories are as close as 12 m from the limit of the property we received from Tantalus. No test pits and drill holes are outside of the property. No resources were modeled outside of the property.

#### 14.11 Inclusion of “Dead Zones” to Constrain the Resource

It was noted that some areas miss 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. To make sure not to estimate material in those areas, some outlines were drawn by hand and the resource of that area was removed from the resource tables. A total of 42 “dead zones” were outlined. The Figure 14-16 shows the outlines for the different “dead zones” along with the collars of the test pits and drill holes.

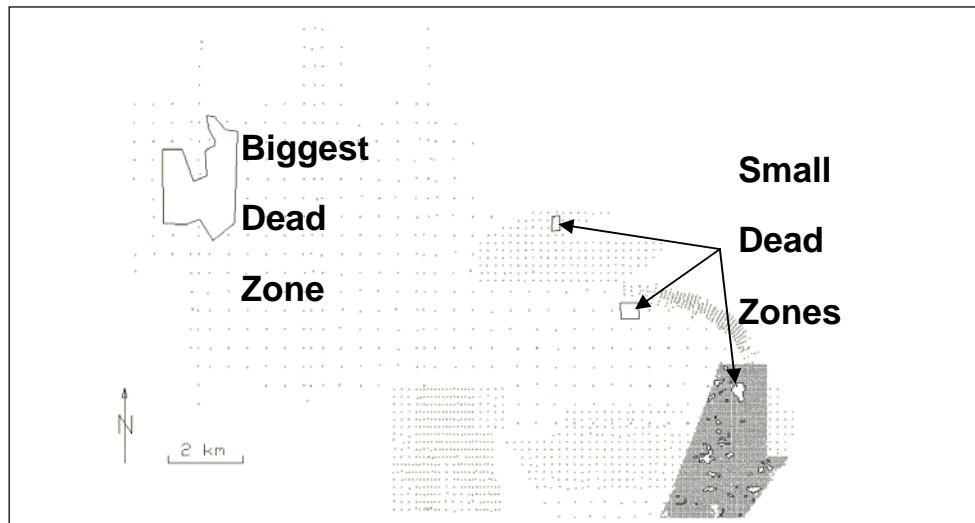


Figure 14-16: Dead Zones Outlines Where Test pits and drill holes are Missing

### 14.12 Delimitation of Sloping Areas vs Flat Areas

Quicksurf software was used to make contours of the flat areas using the selected topography surface. The sloping areas are considered to have a lower COG because the in-situ leaching could be used to extract rare earth elements. The flat areas are considered to have a higher COG because the material would need to be mined and transported to a plant facility. The treatment in the plant would also be at higher costs. Blocks lying in the flat areas have been assigned a COGcode of 2. A total of 225 flat areas were outlined. Flat areas are shown on Figure 14-17.

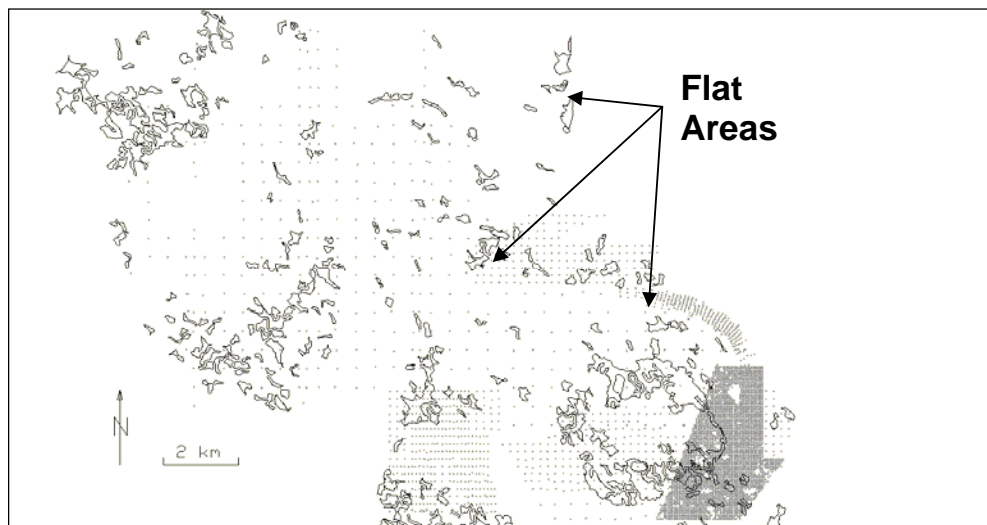


Figure 14-17: Flat Areas Where the Topography Dips Less Than 5 degrees

### 14.13 Separation of the Resource into the Different Prospects

The prospects outlines were provided by the client. Prospect outlines are shown on Figure 14-18.

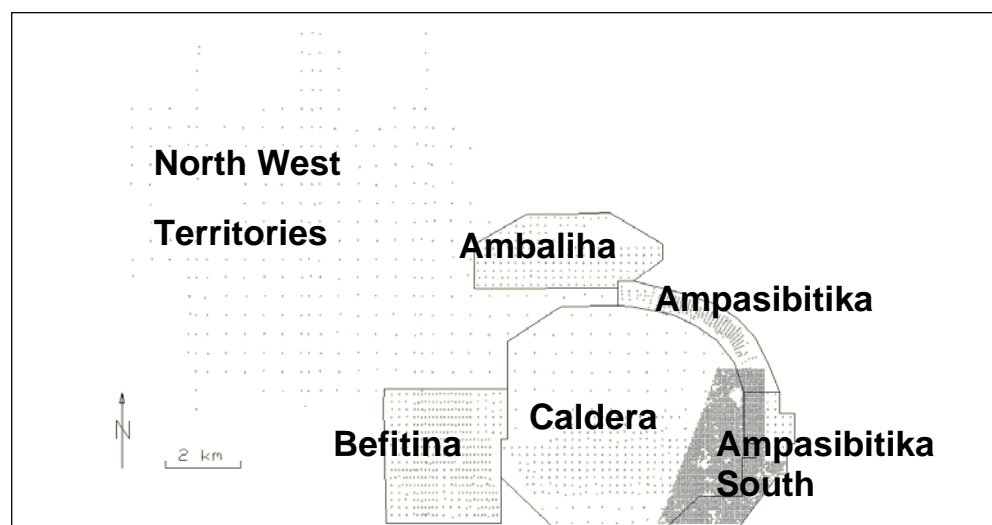


Figure 14-18: Prospects Outlines

### 14.14 Base Case for the Estimation of the Resources

#### 14.14.1 Determination of the Cut-Off Grade for the Base Case of the Resources

In order 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 35% was also applied to the in-situ value to account for last processing stage (REO separation). That is the REE concentrates are assumed to have 65% of the value of the refined metals.

The price list was set by looking at 4 rare earth projects that are at preliminary economic assessment of feasibility stage with public reports from 2013 or 2014. The second source is 2 market analysis groups with prices dated October 14, 2014. Because the market appears to be declining, the market analysis groups prices tend to be lower and we mainly selected these prices as shown in Table 14-8.

Since some recovery estimates are available from metallurgical tests (see Item 13 for more details), we elected to use the recoveries presented in Table 14-9.

At this stage, 2 processing scenarios are considered: in-situ leaching and tank leaching. The in-situ leaching is less expensive to execute per unit metal produced, therefore a lower COG is achieved. Since some material in flat areas can only be mined and treated by tank leaching, a different COG has to be applied to that material. The final determination of the COGs is presented in Table 14-10. The Net Smelter Revenue (NSR) cut-off is shown here, however we elected to apply the cut-off based on element concentrations which is coherent with industry standards and would be more practical to apply in a mining scenario. As Ce and La have little value compared to their concentrations, these

were not considered for the application of the cut off value. The cut-off values are coherent with published values from the analogous Chinese deposits.

**Table 14-8: Rare Earth Oxides Price List Used for the Cut-Off Grade Determination**

Rare Earth Oxide	Source 1 \$/kg	Source 2 \$/kg	Source 3 \$/kg	Source 4 \$/kg	Avg. 1-4 \$/kg	Source 5 \$/kg	Source 6 \$/kg	Avg. 5-6 \$/kg	Chosen \$/kg
Ce <sub>2</sub> O <sub>3</sub>	5.90	8.00	5.00	6.23	6.28	4.75	4.05	4.40	4.40
La <sub>2</sub> O <sub>3</sub>	5.95	9.00	10.00		8.32	8.85	7.50	8.18	8.20
Pr <sub>2</sub> O <sub>3</sub>	75.40	85.00	75.00	63.78	74.80	119.50	115.50	117.50	117.00
Nd <sub>2</sub> O <sub>3</sub>	75.00	80.00	75.00	63.78	73.45	59.50	60.50	60.00	60.00
Sm <sub>2</sub> O <sub>3</sub>	6.85	9.00	10.00		8.62	5.50	3.90	4.70	4.70
Eu <sub>2</sub> O <sub>3</sub>	1,100.00	1,000.00	500.00	1,349.03	987.26	725.00	655.00	690.00	700.00
Gd <sub>2</sub> O <sub>3</sub>	59.40	40.00	40.00	20.13	39.88	46.50	16.89	31.70	32.00
Tb <sub>2</sub> O <sub>3</sub>	1,076.00	950.00	975.00	1,523.63	1,131.16	615.00	555.00	585.00	590.00
Dy <sub>2</sub> O <sub>3</sub>	713.00	650.00	520.00	718.53	650.38	340.00	295.00	317.50	320.00
Ho <sub>2</sub> O <sub>3</sub>	53.60	55.00	NA	16.19	41.60	NA	43.26	43.26	43.00
Er <sub>2</sub> O <sub>3</sub>	63.60	70.00	NA	37.19	56.93	NA	62.50	62.50	60.00
Tm <sub>2</sub> O <sub>3</sub>	1,200.00	1,000.00	NA	81.19	760.40	NA	NA	NA	700.00
Yb <sub>2</sub> O <sub>3</sub>	56.70	50.00	NA	16.19	40.96	NA	31.83	31.83	32.00
Lu <sub>2</sub> O <sub>3</sub>	1,400.00	1,100.00	NA	336.19	945.40	NA	1,216.06	1,216.06	1,200.00
Y <sub>2</sub> O <sub>3</sub>	29.40	30.00	20.00	15.28	23.67	13.30	12.50	12.90	13.00

**Table 14-9: Rare Earth Oxides Recoveries Used for the Cut-Off Grade Determination**

Rare Earth Oxide	Outotec	SGS IS 1	SGS IS 2	SGS IS 3	SGS IS 4	Avg. & Chosen
	%	%	%	%	%	%
Ce <sub>2</sub> O <sub>3</sub>	12.50	13.00	11.00	10.00	10.00	11.63
La <sub>2</sub> O <sub>3</sub>	82.40	85.00	82.00	77.00	71.00	81.60
Pr <sub>2</sub> O <sub>3</sub>	83.30	84.00	82.00	77.00	71.00	81.58
Nd <sub>2</sub> O <sub>3</sub>	76.30	85.00	83.00	78.00	73.00	80.58
Sm <sub>2</sub> O <sub>3</sub>	70.80	81.00	79.00	74.00	69.00	76.20
Eu <sub>2</sub> O <sub>3</sub>	51.90	83.00	80.00	78.00	67.00	73.23
Gd <sub>2</sub> O <sub>3</sub>	54.10	80.00	80.00	76.00	74.00	72.53
Tb <sub>2</sub> O <sub>3</sub>	54.70	75.00	76.00	70.00	68.00	68.93
Dy <sub>2</sub> O <sub>3</sub>	48.60	70.00	68.00	65.00	64.00	62.90
Ho <sub>2</sub> O <sub>3</sub>	43.00	63.00	64.00	62.00	59.00	58.00
Er <sub>2</sub> O <sub>3</sub>	37.20	57.00	58.00	56.00	55.00	52.05
Tm <sub>2</sub> O <sub>3</sub>	30.90	51.00	49.00	47.00	49.00	44.48
Yb <sub>2</sub> O <sub>3</sub>	31.20	49.00	47.00	46.00	47.00	43.30
Lu <sub>2</sub> O <sub>3</sub>	21.50	45.00	45.00	40.00	43.00	37.88
Y <sub>2</sub> O <sub>3</sub>	46.20	64.00	64.00	62.00	60.00	59.05

**Table 14-10: Cut-Off Grade Final Determination**

Items	Units	Values	
		In-situ Leach	Tank Leach
"Mining" cost	US\$/t mined	\$ 0.70	\$ 1.40
Processing cost	US\$/t mined	\$ 5.79	\$ 9.31
G&A	US\$/t mined	\$ 0.75	\$ 0.75
<b>Marginal Cut off NSR</b>	<b>US\$/t mined</b>	<b>\$ 7.24</b>	<b>\$ 11.46</b>
<b>Equivalent TREOnoCe** cut-off</b>	<b>ppm</b>	<b>300</b>	<b>500</b>

\*\* : TREOnoCe includes Y<sub>2</sub>O<sub>3</sub> but excludes Ce<sub>2</sub>O<sub>3</sub>. It is a good indicator of the value of the material.

### 14.14.2 Resource Tables for the Base Case

The base case resource with tonnage of the mineral and grade is presented in the Table 14-11. The details for each prospect and for flat and sloping areas are in Table 14-12. The full details for each of the rare earth elements are in Table 14-13. The plan view of the blocks summed in the base case is shown in Figure 14-19.

**Table 14-11: Resource Base Case – COGs of 300 and 500 ppm TREOnoCe**

Classification	Tonnage (t)	Volume (m <sup>3</sup> )	Area (m <sup>2</sup> )	Density (t/m <sup>3</sup> )	Thickness (m)			TREO (ppm)	TREOnoCe (ppm)	CREO (ppm)	HREO (ppm)	LREO (ppm)	HREO / TREOnoCe ratio	Contained TREO * (t)
					Total	PED	SAP							
Measured	40,103,550	35,948,700	6,618,600	1.12	5.4	2.8	2.6	975	660	296	187	788	28%	39,092
Indicated	157,580,640	143,150,400	20,998,800	1.10	6.8	2.6	4.2	878	554	255	166	712	30%	138,292
<b>Measured + Indicated</b>	<b>197,684,190</b>	<b>179,099,100</b>	<b>27,617,400</b>	<b>1.10</b>	<b>6.5</b>	<b>2.7</b>	<b>3.8</b>	<b>897</b>	<b>575</b>	<b>263</b>	<b>170</b>	<b>727</b>	<b>30%</b>	<b>177,383</b>
Inferred	429,999,525	390,900,600	70,396,200	1.10	5.6	2.7	2.9	894	574	247	149	745	26%	384,552

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price

- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

\* Contained TREO is presented as in-situ. Values do not account for recovery losses.

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

CREO = Nd<sub>2</sub>O<sub>3</sub>+Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

Table 14-12: Base Case – Resource Details by Prospect

Classification	Prospect	Area Type *	Tonnage (t)	Volume (m <sup>3</sup> )	Area (m <sup>2</sup> )	Density (t/m <sup>3</sup> )	Thickness (m)			TREO (ppm)	TREOnoCe (ppm)	CREO (ppm)	HREO (ppm)	LREO (ppm)	HREO / TREOnoCe ratio	Contained TREO **** (t)
							Total	PED	SAP							
Measured	Various **	sloping	38,023,000	34,084,000	6,345,000	1.12	5.4	2.8	2.6	972	657	295	186	786	28%	36,973
	Various ***	flat	2,081,000	1,865,000	274,000	1.12	6.8	3.0	3.9	1,018	709	324	206	812	29%	2,119
	<b>TOTAL</b>	<b>TOTAL</b>		<b>40,104,000</b>	<b>35,949,000</b>	<b>6,619,000</b>	<b>1.12</b>	<b>5.4</b>	<b>2.8</b>	<b>2.6</b>	<b>975</b>	<b>660</b>	<b>296</b>	<b>187</b>	<b>788</b>	28%
Indicated	Ambaliha	sloping	22,553,000	20,503,000	3,937,000	1.10	5.2	2.1	3.1	752	457	214	144	609	31%	16,970
		flat	338,000	307,000	64,000	1.10	4.8	1.7	3.1	949	632	276	172	777	27%	320
	Amipasitika	sloping	25,780,000	23,436,000	1,871,000	1.10	12.5	5.2	7.3	899	590	275	187	711	32%	23,164
		flat	143,000	130,000	18,000	1.10	7.2	2.0	5.2	899	677	323	209	690	31%	128
	Amipasitika S.	sloping	2,650,000	2,304,000	631,000	1.15	3.7	2.8	0.8	613	377	177	116	497	31%	1,624
	Befitina	sloping	50,487,000	45,897,000	7,087,000	1.10	6.5	2.9	3.5	894	540	251	164	729	30%	45,112
		flat	1,067,000	970,000	153,000	1.10	6.3	3.1	3.3	937	617	284	176	761	29%	1,000
	Caldera	sloping	50,722,000	46,111,000	6,763,000	1.10	6.8	1.9	4.9	915	593	266	168	747	28%	46,398
		flat	3,599,000	3,272,000	438,000	1.10	7.5	2.5	5.0	933	627	287	183	750	29%	3,359
	North West T.	sloping	200,000	182,000	32,000	1.10	5.6	2.3	3.3	768	492	220	139	629	28%	154
		flat	44,000	40,000	5,000	1.10	7.3	2.2	5.2	775	536	241	150	625	28%	34
	<b>TOTAL</b>	<b>TOTAL</b>		<b>157,581,000</b>	<b>143,150,000</b>	<b>20,999,000</b>	<b>1.10</b>	<b>6.8</b>	<b>2.6</b>	<b>4.2</b>	<b>877</b>	<b>554</b>	<b>254</b>	<b>166</b>	<b>711</b>	<b>30%</b>
<b>Measured + Indicated</b>	<b>TOTAL</b>	<b>TOTAL</b>	<b>197,684,000</b>	<b>179,099,000</b>	<b>27,617,000</b>	<b>1.10</b>	<b>6.5</b>	<b>2.7</b>	<b>3.8</b>	<b>897</b>	<b>575</b>	<b>263</b>	<b>170</b>	<b>727</b>	<b>30%</b>	<b>177,353</b>
Inferred	Ambaliha	sloping	74,000	68,000	10,000	1.10	6.8	2.9	3.9	826	597	283	179	647	30%	61
	Amipasitika	sloping	3,337,000	3,034,000	345,000	1.10	8.8	3.3	5.5	902	610	279	182	720	30%	3,010
		flat	70,000	64,000	5,000	1.10	11.8	3.7	8.2	861	585	265	174	687	30%	61
	Amipasitika S.	sloping	204,000	177,000	59,000	1.15	3.0	1.7	1.3	727	465	211	128	599	28%	148
	Befitina	sloping	1,755,000	1,596,000	296,000	1.10	5.4	2.3	3.1	918	590	262	152	765	26%	1,611
		flat	31,000	28,000	4,000	1.10	7.8	2.8	5.0	1,193	842	359	192	1,002	23%	37
	Caldera	sloping	90,499,000	82,272,000	13,598,000	1.10	6.1	2.4	3.6	847	531	236	146	701	27%	76,634
		flat	6,683,000	6,076,000	734,000	1.10	8.3	2.7	5.5	880	573	252	148	732	26%	5,881
	North West T.	sloping	314,212,000	285,647,000	53,677,000	1.10	5.3	2.7	2.6	908	584	250	150	757	26%	285,207
		flat	13,133,000	11,939,000	1,669,000	1.10	7.2	4.7	2.5	906	600	245	143	763	24%	11,901
<b>TOTAL</b>	<b>TOTAL</b>		<b>430,000,000</b>	<b>390,901,000</b>	<b>70,396,000</b>	<b>1.10</b>	<b>5.6</b>	<b>2.7</b>	<b>2.9</b>	<b>894</b>	<b>574</b>	<b>247</b>	<b>149</b>	<b>745</b>	<b>26%</b>	<b>384,551</b>

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

\* Area Type can be sloping (sloping greater than 5 degrees) or flat. Sloping areas can potentially be mined by on site leaching, requiring lower costs.

\*\* Amipasitika, Amipasitika South and Caldera

\*\*\* Amipasitika South and Caldera

\*\*\*\* Contained TREO is presented as in-situ. Values do not account for recovery losses.

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

CREO = Nd<sub>2</sub>O<sub>3</sub>+Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

Table 14-13: Base Case – Resource Details by Prospect with individual REOs

Classification	Prospect	Area Type *	Tonnage (t)	Y <sub>2</sub> O <sub>3</sub> (ppm)	La <sub>2</sub> O <sub>3</sub> (ppm)	Ce <sub>2</sub> O <sub>3</sub> (ppm)	Pr <sub>2</sub> O <sub>3</sub> (ppm)	Nd <sub>2</sub> O <sub>3</sub> (ppm)	Sm <sub>2</sub> O <sub>3</sub> (ppm)	Eu <sub>2</sub> O <sub>3</sub> (ppm)	Gd <sub>2</sub> O <sub>3</sub> (ppm)	Tb <sub>2</sub> O <sub>3</sub> (ppm)	Dy <sub>2</sub> O <sub>3</sub> (ppm)	Ho <sub>2</sub> O <sub>3</sub> (ppm)	Er <sub>2</sub> O <sub>3</sub> (ppm)	Tm <sub>2</sub> O <sub>3</sub> (ppm)	Yb <sub>2</sub> O <sub>3</sub> (ppm)	Lu <sub>2</sub> O <sub>3</sub> (ppm)	Nb <sub>2</sub> O <sub>5</sub> (ppm)	Ta <sub>2</sub> O <sub>5</sub> (ppm)	ThO <sub>2</sub> (ppm)	U <sub>3</sub> O <sub>8</sub> (ppm)
Measured	Various **	sloping	38,023,000	112	240	315	46.6	158	26.9	2.8	22.9	3.3	19.0	3.6	10.3	1.4	9.2	1.4	219	12.4	48.3	9.3
	Various ***	flat	2,081,000	125	251	310	50.4	171	30.0	3.1	25.5	3.7	21.1	3.9	11.1	1.5	9.7	1.4	197	11.2	53.5	10.0
	<b>TOTAL</b>	<b>TOTAL</b>	<b>40,104,000</b>	<b>113</b>	<b>241</b>	<b>315</b>	<b>46.8</b>	<b>158</b>	<b>27.0</b>	<b>2.9</b>	<b>23.0</b>	<b>3.3</b>	<b>19.1</b>	<b>3.6</b>	<b>10.4</b>	<b>1.4</b>	<b>9.2</b>	<b>1.4</b>	<b>218</b>	<b>12.4</b>	<b>48.6</b>	<b>9.4</b>
Indicated	Ambaliha	sloping	22,553,000	86	154	295	31.7	108	19.6	2.7	15.5	2.4	14.5	2.9	8.5	1.3	8.1	1.2	151	9.8	50.4	9.7
		flat	338,000	103	239	317	45.2	150	26.0	2.9	19.9	2.9	17.4	3.4	9.9	1.4	9.1	1.3	195	12.2	51.6	9.7
	Ampasibitika	sloping	25,780,000	115	202	309	40.2	136	24.7	3.0	20.3	3.2	18.7	3.7	10.7	1.6	9.7	1.4	191	11.7	55.7	11.1
		flat	143,000	124	222	222	47.3	169	29.9	5.2	26.2	3.9	21.6	4.2	11.8	1.6	9.9	1.5	78	4.9	38.1	7.2
	Ampasibitika S.	sloping	2,650,000	70	128	235	26.2	91	16.1	2.2	13.0	2.0	11.6	2.3	6.7	1.0	6.2	0.9	122	7.2	37.4	6.9
	Befitina	sloping	50,487,000	100	187	353	37.6	129	22.8	2.8	17.4	2.7	16.6	3.3	9.6	1.4	9.2	1.4	237	13.6	56.0	11.2
		flat	1,067,000	105	216	320	44.2	154	27.1	3.8	19.9	3.0	18.0	3.5	10.1	1.5	9.5	1.4	213	12.3	51.2	10.2
	Caldera	sloping	50,722,000	103	218	321	42.4	141	24.5	2.4	18.5	2.8	16.8	3.3	9.5	1.4	8.8	1.3	246	14.0	52.3	10.5
		flat	3,599,000	112	221	307	44.6	150	27.5	2.4	21.1	3.2	18.7	3.6	10.0	1.5	9.1	1.4	230	13.4	53.3	11.1
	North West T.	sloping	200,000	83	179	277	35.3	118	20.6	2.9	16.7	2.4	14.1	2.7	7.8	1.1	7.2	1.1	174	9.8	39.6	7.7
		flat	44,000	89	195	239	39.0	130	22.4	3.5	18.8	2.7	15.3	2.9	8.1	1.2	7.4	1.1	139	8.3	37.8	7.6
	<b>TOTAL</b>	<b>TOTAL</b>	<b>157,581,000</b>	<b>101</b>	<b>195</b>	<b>324</b>	<b>38.8</b>	<b>131</b>	<b>23.2</b>	<b>2.7</b>	<b>18.0</b>	<b>2.8</b>	<b>16.7</b>	<b>3.3</b>	<b>9.6</b>	<b>1.4</b>	<b>8.9</b>	<b>1.3</b>	<b>218</b>	<b>12.7</b>	<b>53.5</b>	<b>10.7</b>
<b>Measured + Indicated</b>	<b>TOTAL</b>	<b>TOTAL</b>	<b>197,684,000</b>	<b>104</b>	<b>204</b>	<b>322</b>	<b>40.4</b>	<b>137</b>	<b>24.0</b>	<b>2.7</b>	<b>19.0</b>	<b>2.9</b>	<b>17.2</b>	<b>3.4</b>	<b>9.7</b>	<b>1.4</b>	<b>9.0</b>	<b>1.3</b>	<b>218</b>	<b>12.7</b>	<b>52.5</b>	<b>10.4</b>
Inferred	Ambaliha	sloping	74,000	103	196	229	42.6	153	26.8	4.7	22.4	3.3	18.8	3.6	10.4	1.4	9.2	1.3	74	4.6	38.7	7.3
	Ampasibitika	sloping	3,337,000	110	216	292	42.3	144	25.4	3.0	20.8	3.2	18.6	3.6	10.4	1.5	9.4	1.4	191	10.9	47.1	9.2
		flat	70,000	106	211	276	40.3	135	23.8	2.7	19.1	3.0	17.6	3.5	9.9	1.5	9.3	1.4	201	11.6	43.1	9.1
	Ampasibitika S.	sloping	204,000	75	165	262	33.8	118	20.2	2.8	16.1	2.3	13.2	2.5	7.2	1.0	6.8	1.0	116	6.8	42.2	7.6
	Befitina	sloping	1,755,000	92	220	328	43.7	149	24.7	2.9	18.0	2.6	15.4	3.0	8.5	1.2	7.9	1.2	241	12.8	40.0	8.2
		flat	31,000	115	333	352	64.5	218	34.3	3.8	22.5	3.3	19.2	3.7	11.0	1.6	10.5	1.6	279	14.9	47.1	10.0
	Caldera	sloping	90,499,000	88	197	316	38.5	128	21.8	2.5	16.2	2.4	14.7	2.9	8.5	1.2	7.9	1.2	247	13.5	44.4	8.9
		flat	6,683,000	89	215	307	42.8	143	24.1	2.6	17.1	2.5	15.1	2.9	8.5	1.2	7.8	1.2	227	12.8	46.7	9.3
	North West T.	sloping	314,212,000	91	230	323	42.6	140	22.0	2.4	18.0	2.6	14.9	2.9	8.5	1.2	8.0	1.2	219	13.1	43.0	8.9
		flat	13,133,000	86	252	306	43.8	140	20.9	2.2	16.9	2.4	14.0	2.7	8.2	1.2	7.9	1.2	236	14.5	41.2	8.6
<b>TOTAL</b>	<b>TOTAL</b>	<b>430,000,000</b>	<b>90</b>	<b>223</b>	<b>321</b>	<b>41.8</b>	<b>137</b>	<b>22.0</b>	<b>2.4</b>	<b>17.6</b>	<b>2.6</b>	<b>14.8</b>	<b>2.9</b>	<b>8.5</b>	<b>1.2</b>	<b>8.0</b>	<b>1.2</b>	<b>225</b>	<b>13.2</b>	<b>43.3</b>	<b>8.9</b>	

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

\* Area Type can be sloping (sloping greater than 5 degrees) or flat. Sloping areas can potentially be mined by on site leaching, requiring lower costs.

\*\* Ampasibitika, Ampasibitika South and Caldera

\*\*\* Ampasibitika South and Caldera

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

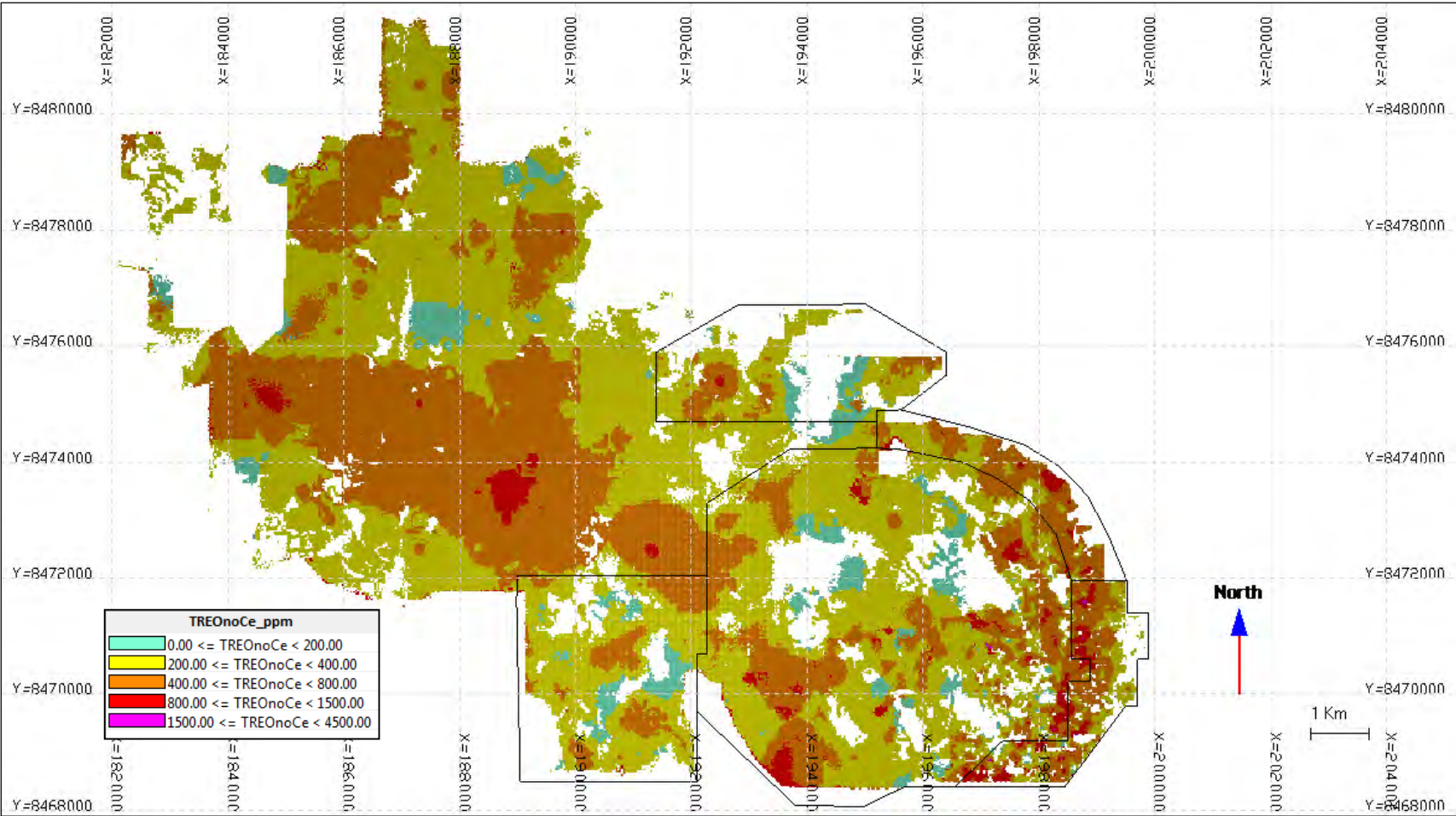


Figure 14-19: Plan view of the blocks included in the base case resource

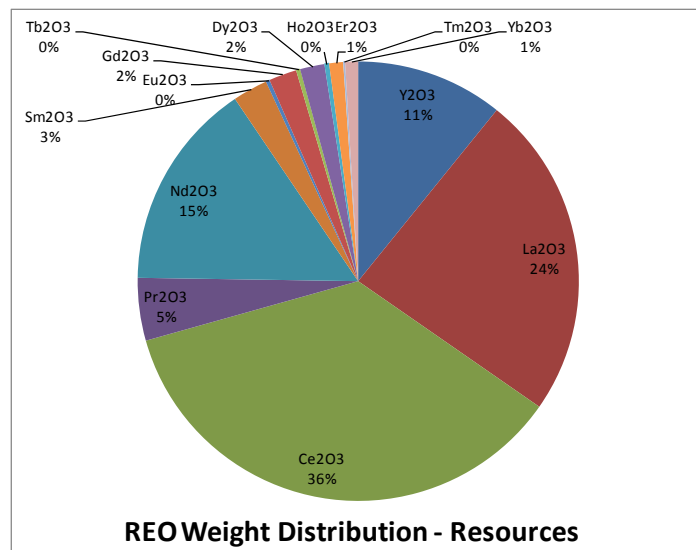
**14.14.3 Distribution of the Oxides Contained in the Resources, in the Concentrate, and Potential Value for the Base Case**

The Table 14-14 shows the tonnage of the individual oxides contained in the project mineral resources. Contained TREO is presented as in-situ. Values do not account for recovery losses. The Figure 14-20 shows the distribution of these tonnages. Because each REO has a different recovery stated in Table 14-9, the actual tonnages of the individual oxides that can potentially be contained in the concentrates would be less. These tonnages contained in the potential concentrates are shown in Table 14-15 and their distribution is shown in Figure 14-21. Now since each REO has a different value that is controlled by the market prices stated in Table 14-8, the distribution of the potential revenues generated by each REO would be very different. The distribution of the potential revenues generated by the individual oxides is shown in Figure 14-22. Gross values are not reported as recommended by NI 43-101 standards. It has to be noted that the distributions for measured, indicated and inferred resources are always the same +/- 1%. Therefore only the average distributions are shown in figures.

**Table 14-14: Tonnage of the Individual Oxides Contained in the Project Mineral Resources**

Classification	Y <sub>2</sub> O <sub>3</sub> (t)	La <sub>2</sub> O <sub>3</sub> (t)	Ce <sub>2</sub> O <sub>3</sub> (t)	Pr <sub>2</sub> O <sub>3</sub> (t)	Nd <sub>2</sub> O <sub>3</sub> (t)	Sm <sub>2</sub> O <sub>3</sub> (t)	Eu <sub>2</sub> O <sub>3</sub> (t)	Gd <sub>2</sub> O <sub>3</sub> (t)	Tb <sub>2</sub> O <sub>3</sub> (t)	Dy <sub>2</sub> O <sub>3</sub> (t)	Ho <sub>2</sub> O <sub>3</sub> (t)	Er <sub>2</sub> O <sub>3</sub> (t)	Tm <sub>2</sub> O <sub>3</sub> (t)	Yb <sub>2</sub> O <sub>3</sub> (t)	Lu <sub>2</sub> O <sub>3</sub> (t)	TREO (t)
Measured	4,520	9,667	12,620	1,876	6,347	1,084	115	923	133	766	144	415	58	369	55	39,092
Indicated	15,953	30,677	51,031	6,110	20,659	3,661	423	2,837	440	2,630	522	1,507	222	1,409	210	138,292
<b>Measured + Indicated</b>	<b>20,472</b>	<b>40,344</b>	<b>63,651</b>	<b>7,986</b>	<b>27,006</b>	<b>4,745</b>	<b>538</b>	<b>3,760</b>	<b>573</b>	<b>3,397</b>	<b>666</b>	<b>1,922</b>	<b>280</b>	<b>1,779</b>	<b>265</b>	<b>177,383</b>
Inferred	38,745	95,894	137,928	17,960	59,110	9,468	1,038	7,578	1,097	6,384	1,235	3,645	521	3,431	517	384,552

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.  
 - The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees  
 - The cut-off grade is 500 ppm TREOnoCe for flat areas  
 Contained TREO is presented as in-situ. Values do not account for recovery losses.  
 TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>  
 HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>  
 LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>



**Figure 14-20: Distribution of the Individual Oxides Contained in the Project Mineral Resources**

**Table 14-15: Tonnage of the Individual Oxides Contained in the Potential Concentrates**

Classification	Y <sub>2</sub> O <sub>3</sub> (t)	La <sub>2</sub> O <sub>3</sub> (t)	Ce <sub>2</sub> O <sub>3</sub> (t)	Pr <sub>2</sub> O <sub>3</sub> (t)	Nd <sub>2</sub> O <sub>3</sub> (t)	Sm <sub>2</sub> O <sub>3</sub> (t)	Eu <sub>2</sub> O <sub>3</sub> (t)	Gd <sub>2</sub> O <sub>3</sub> (t)	Tb <sub>2</sub> O <sub>3</sub> (t)	Dy <sub>2</sub> O <sub>3</sub> (t)	Ho <sub>2</sub> O <sub>3</sub> (t)	Er <sub>2</sub> O <sub>3</sub> (t)	Tm <sub>2</sub> O <sub>3</sub> (t)	Yb <sub>2</sub> O <sub>3</sub> (t)	Lu <sub>2</sub> O <sub>3</sub> (t)	TREO (t)
Measured	2,677	7,683	1,426	1,491	5,018	811	83	672	91	484	84	219	26	163	21	20,948
Indicated	9,450	24,382	5,767	4,855	16,333	2,737	305	2,066	302	1,660	304	793	101	621	82	69,757
<b>Measured + Indicated</b>	<b>12,128</b>	<b>32,065</b>	<b>7,193</b>	<b>6,346</b>	<b>21,351</b>	<b>3,548</b>	<b>387</b>	<b>2,738</b>	<b>394</b>	<b>2,144</b>	<b>388</b>	<b>1,012</b>	<b>127</b>	<b>783</b>	<b>103</b>	<b>90,705</b>
Inferred	22,953	76,217	15,586	14,271	46,732	7,079	747	5,518	754	4,030	719	1,919	236	1,511	201	198,473

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

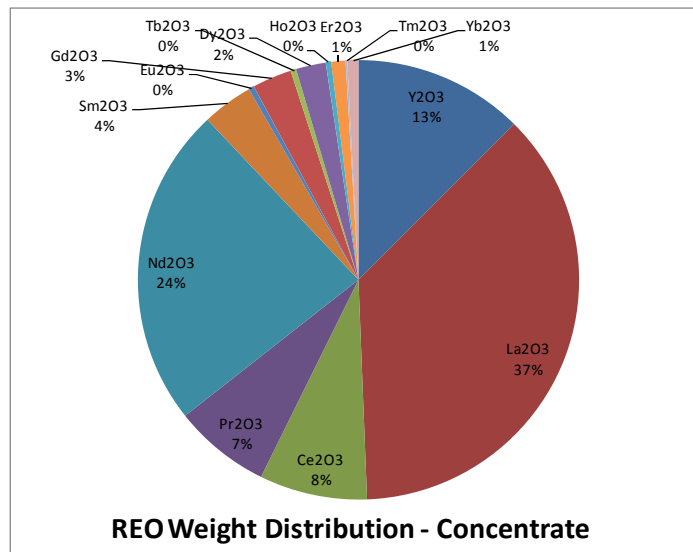
- The cut-off grade is 300 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 500 ppm TREOnoCe for flat areas

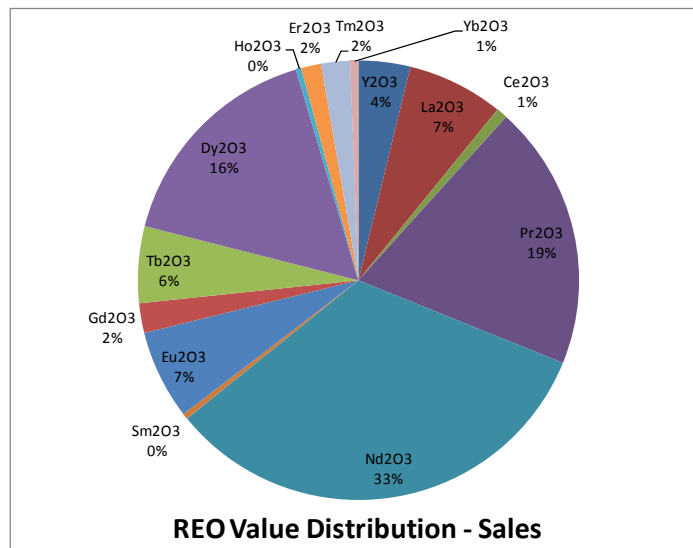
TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>



**Figure 14-21: Distribution of the Individual Oxides Contained in the Potential Concentrates**



**Figure 14-22: Distribution of the Potential Revenues Generated by the Individual Oxides**

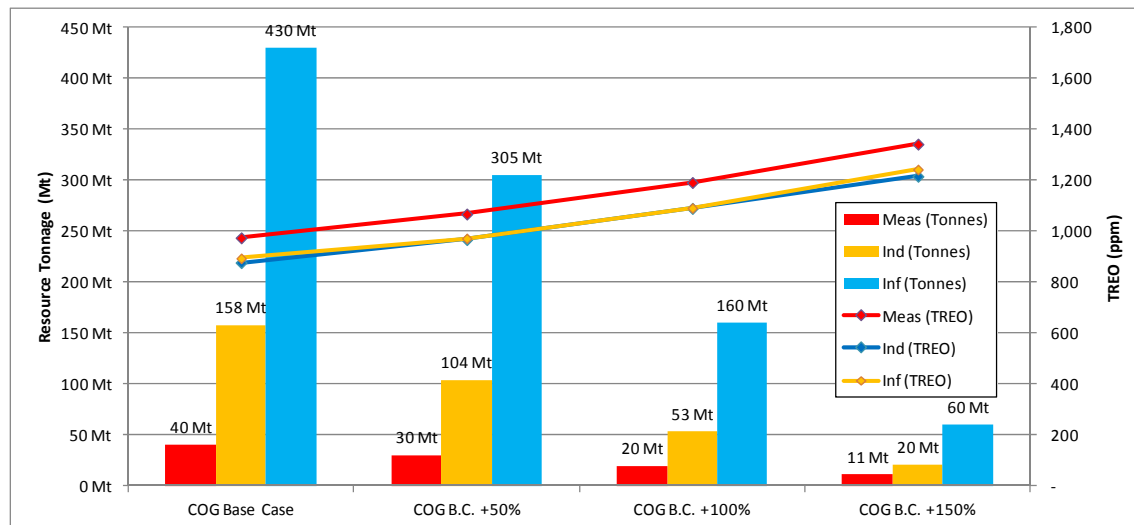
### 14.15 Estimation of the Resources – Sensitivity Analysis

In order to evaluate the sensitivity of the deposit to a reduction of the rare earth prices or to an increase in production costs, we estimated the resources with different cut-off grades. We chose to evaluate the resources with cut-off grades 50%, 100% and 150% higher as shown in Table 14-16. An increase of the cut-off grade by 100% could be the result of rare earth prices cut in half, or the result of the production costs facing a twofold increase, or a combination of the rare earth prices decreasing by 30% while production costs increase by 40%. The details of these resources are shown in Table 14-17 and Table 14-18. The Figure 14-23 is the resulting TREO grade – tonnage curves for measured, indicated and inferred resources. At all cut-off grades, it is interesting that the HREO/TREOnoCe ratio stays between 22% and 32%. Also the TREOnoCe/TREO ratio stays between 63% and 73%.

**Table 14-16: Cut-Off Grades Used for the Sensitivity Analysis**

	COG*	
	sloping Areas	Flat Areas
Base Case	300	500
Base Case +50%	450	750
Base Case +100%	600	1000
Base Case +150%	750	1250

\* Cut-Off Grade is on TREOnoCe (ppm)



**Figure 14-23: Tonnage and TREO Grade Variation with Different Cut-Off Grades**

**Table 14-17: Base Case +100% – Resource Details by Prospect**

Classification	Prospect	Area Type *	Tonnage (t)	Volume (m <sup>3</sup> )	Area (m <sup>2</sup> )	Density (t/m <sup>3</sup> )	Thickness (m)			TREO (ppm)	TREOnoCe (ppm)	CREO (ppm)	HREO (ppm)	LREO (ppm)	HREO / TREOnoCe ratio	Contained TREO **** (t)
							Total	PED	SAP							
Measured	Various **	sloping	19,634,000	17,587,000	3,138,000	1.12	5.6	2.8	2.8	1,189	840	371	228	960	27%	23,340
	Various ***	flat	193,000	170,000	27,000	1.14	6.3	2.4	3.9	1,487	1,176	517	316	1,171	27%	287
	<b>TOTAL</b>	<b>TOTAL</b>		<b>19,827,000</b>	<b>17,757,000</b>	<b>3,165,000</b>	<b>1.12</b>	<b>5.6</b>	<b>2.8</b>	<b>2.8</b>	<b>1,192</b>	<b>844</b>	<b>373</b>	<b>229</b>	<b>963</b>	27%
Indicated	Ambaliha	sloping	2,838,000	2,580,000	470,000	1.10	5.5	1.9	3.6	1,087	703	303	187	900	27%	3,085
	Ampasibitika	sloping	11,333,000	10,302,000	800,000	1.10	12.9	5.0	7.9	1,063	718	334	225	837	31%	12,042
	Ampasibitika S.	sloping	30,000	26,000	14,000	1.15	1.9	1.5	0.4	963	645	295	177	786	27%	29
	Befitina	sloping	16,204,000	14,731,000	2,155,000	1.10	6.8	2.6	4.3	1,101	748	339	210	891	28%	17,838
		flat	24,000	22,000	3,000	1.10	8.0	1.3	6.7	1,318	1,026	451	254	1,064	25%	31
	Caldera	sloping	22,821,000	20,747,000	3,117,000	1.10	6.7	1.7	5.0	1,097	742	322	191	906	26%	25,027
		flat	6,000	5,000	1,000	1.10	6.0	2.0	4.0	1,453	1,088	440	230	1,222	21%	9
	<b>TOTAL</b>	<b>TOTAL</b>		<b>53,256,000</b>	<b>48,414,000</b>	<b>6,558,000</b>	<b>1.10</b>	<b>7.4</b>	<b>2.4</b>	<b>5.0</b>	<b>1,090</b>	<b>737</b>	<b>329</b>	<b>204</b>	<b>887</b>	28%
<b>Measured + Indicated</b>	<b>TOTAL</b>	<b>TOTAL</b>	<b>73,084,000</b>	<b>66,171,000</b>	<b>9,724,000</b>	<b>1.10</b>	<b>6.8</b>	<b>2.5</b>	<b>4.3</b>	<b>1,118</b>	<b>766</b>	<b>341</b>	<b>211</b>	<b>907</b>	<b>28%</b>	<b>81,689</b>
Inferred	Ambaliha	sloping	41,000	37,000	4,000	1.10	10.3	4.5	5.8	883	658	312	197	686	30%	36
	Ampasibitika	sloping	2,080,000	1,891,000	187,000	1.10	10.1	3.2	6.9	982	678	309	201	781	30%	2,042
	Ampasibitika S.	sloping	43,000	38,000	13,000	1.15	3.0	1.3	1.7	1,011	709	311	173	837	24%	44
	Befitina	sloping	787,000	716,000	105,000	1.10	6.8	2.5	4.2	1,109	727	318	187	921	26%	873
	Caldera	sloping	23,330,000	21,209,000	3,037,000	1.10	7.0	1.7	5.3	1,106	753	327	194	912	26%	25,804
	North West T.	sloping	133,419,000	121,290,000	19,646,000	1.10	6.2	2.7	3.4	1,091	743	317	186	905	25%	145,564
	<b>TOTAL</b>	<b>TOTAL</b>		<b>159,701,000</b>	<b>145,181,000</b>	<b>22,992,000</b>	<b>1.10</b>	<b>6.3</b>	<b>2.6</b>	<b>3.7</b>	<b>1,092</b>	<b>743</b>	<b>319</b>	<b>187</b>	<b>904</b>	25%

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

- The cut-off grade is 600 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 1000 ppm TREOnoCe for flat areas

\* Area Type can be sloping (sloping greater than 5 degrees) or flat. Sloping areas can potentially be mined by on site leaching, requiring lower costs.

\*\* Ampasibitika, Ampasibitika South and Caldera

\*\*\* Ampasibitika South and Caldera

\*\*\*\* Contained TREO is presented as in-situ. Values do not account for recovery losses.

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

CREO = Nd<sub>2</sub>O<sub>3</sub>+Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

**Table 14-18: Base Case +100% – Resource Details by Prospect with individual REOs**

Classification	Prospect	Area Type *	Tonnage (t)	Y <sub>2</sub> O <sub>3</sub> (ppm)	La <sub>2</sub> O <sub>3</sub> (ppm)	Ce <sub>2</sub> O <sub>3</sub> (ppm)	Pr <sub>2</sub> O <sub>3</sub> (ppm)	Nd <sub>2</sub> O <sub>3</sub> (ppm)	Sm <sub>2</sub> O <sub>3</sub> (ppm)	Eu <sub>2</sub> O <sub>3</sub> (ppm)	Gd <sub>2</sub> O <sub>3</sub> (ppm)	Tb <sub>2</sub> O <sub>3</sub> (ppm)	Dy <sub>2</sub> O <sub>3</sub> (ppm)	Ho <sub>2</sub> O <sub>3</sub> (ppm)	Er <sub>2</sub> O <sub>3</sub> (ppm)	Tm <sub>2</sub> O <sub>3</sub> (ppm)	Yb <sub>2</sub> O <sub>3</sub> (ppm)	Lu <sub>2</sub> O <sub>3</sub> (ppm)	Nb <sub>2</sub> O <sub>5</sub> (ppm)	Ta <sub>2</sub> O <sub>5</sub> (ppm)	ThO <sub>2</sub> (ppm)	U <sub>3</sub> O <sub>8</sub> (ppm)
Measured	Various **	sloping	19,634,000	137	314	348	60.4	203	34.3	3.5	29.1	4.1	23.5	4.4	12.4	1.7	10.8	1.6	251	14.2	52.2	10.2
	Various ***	flat	193,000	191	444	311	84.3	283	49.3	5.5	42.1	6.0	32.3	5.9	16.2	2.1	12.9	1.8	205	11.1	51.0	9.7
	<b>TOTAL</b>	<b>TOTAL</b>	<b>19,827,000</b>	<b>138</b>	<b>315</b>	<b>348</b>	<b>60.6</b>	<b>204</b>	<b>34.5</b>	<b>3.5</b>	<b>29.2</b>	<b>4.2</b>	<b>23.6</b>	<b>4.4</b>	<b>12.5</b>	<b>1.7</b>	<b>10.9</b>	<b>1.6</b>	<b>251</b>	<b>14.1</b>	<b>52.2</b>	<b>10.2</b>
Indicated	Ambaliha	sloping	2,838,000	113	271	384	50.4	165	29.0	2.7	22.0	3.2	19.0	3.7	10.7	1.5	9.7	1.4	241	15.0	60.6	10.3
	Ampasibitika	sloping	11,333,000	138	247	345	49.3	166	30.2	3.5	24.8	3.9	22.7	4.5	12.9	1.9	11.5	1.7	223	13.9	63.1	12.5
	Ampasibitika S.	sloping	30,000	105	227	319	47.0	165	28.7	3.6	22.4	3.2	18.7	3.5	9.6	1.3	8.5	1.2	148	8.6	43.2	8.2
	Befitina	sloping	16,204,000	126	269	353	53.8	184	32.3	4.0	24.5	3.7	21.6	4.2	11.7	1.7	10.6	1.6	235	13.4	55.9	11.1
		flat	24,000	152	387	293	76.9	262	45.7	4.9	34.4	5.0	27.0	4.8	12.6	1.7	10.4	1.5	232	13.8	53.9	11.4
	Caldera	sloping	22,821,000	116	286	355	54.6	180	30.5	3.0	22.4	3.3	19.4	3.8	10.5	1.5	9.6	1.4	291	16.2	53.0	10.9
		flat	6,000	138	460	365	82.2	271	44.6	3.5	30.4	4.3	23.4	4.4	11.7	1.8	10.7	1.6	296	18.0	57.5	12.4
<b>TOTAL</b>	<b>TOTAL</b>	<b>53,256,000</b>	<b>124</b>	<b>271</b>	<b>354</b>	<b>53.0</b>	<b>178</b>	<b>30.9</b>	<b>3.4</b>	<b>23.5</b>	<b>3.6</b>	<b>20.7</b>	<b>4.0</b>	<b>11.4</b>	<b>1.6</b>	<b>10.3</b>	<b>1.5</b>	<b>257</b>	<b>14.8</b>	<b>56.4</b>	<b>11.3</b>	
<b>Measured + Indicated</b>	<b>TOTAL</b>	<b>TOTAL</b>	<b>73,084,000</b>	<b>127</b>	<b>283</b>	<b>352</b>	<b>55.1</b>	<b>185</b>	<b>31.9</b>	<b>3.4</b>	<b>25.1</b>	<b>3.7</b>	<b>21.5</b>	<b>4.1</b>	<b>11.7</b>	<b>1.7</b>	<b>10.5</b>	<b>1.5</b>	<b>255</b>	<b>14.6</b>	<b>55.3</b>	<b>11.0</b>
Inferred	Ambaliha	sloping	41,000	114	216	226	46.9	169	29.5	5.2	24.9	3.7	20.8	4.0	11.4	1.6	9.9	1.4	71	4.4	38.9	7.4
	Ampasibitika	sloping	2,080,000	121	241	304	47.2	160	28.3	3.3	23.2	3.6	20.6	4.0	11.5	1.6	10.3	1.5	208	11.7	49.0	9.6
	Ampasibitika S.	sloping	43,000	98	263	302	53.7	187	31.6	4.1	24.2	3.4	18.6	3.5	9.8	1.4	9.0	1.3	148	8.6	48.3	8.8
	Befitina	sloping	787,000	114	277	382	53.5	179	30.1	2.9	22.3	3.3	19.0	3.6	10.3	1.5	9.4	1.4	281	15.1	49.0	10.0
	Caldera	sloping	23,330,000	117	290	353	55.4	183	30.8	3.4	23.2	3.4	19.8	3.8	10.8	1.5	9.8	1.5	295	16.1	48.0	9.7
	North West T.	sloping	133,419,000	112	293	348	54.9	180	28.5	3.0	23.3	3.3	18.5	3.5	10.2	1.4	9.3	1.4	233	14.1	46.4	9.7
	<b>TOTAL</b>	<b>TOTAL</b>	<b>159,701,000</b>	<b>113</b>	<b>292</b>	<b>349</b>	<b>54.9</b>	<b>181</b>	<b>28.9</b>	<b>3.0</b>	<b>23.3</b>	<b>3.3</b>	<b>18.7</b>	<b>3.5</b>	<b>10.3</b>	<b>1.4</b>	<b>9.4</b>	<b>1.4</b>	<b>242</b>	<b>14.4</b>	<b>46.7</b>	<b>9.7</b>

- The cut-off grade is applied to TREOnoCe because it has good correlation with the material value. Ce has high grades but low recovery and market price.

- The cut-off grade is 600 ppm TREOnoCe for areas sloping greater than 5 degrees

- The cut-off grade is 1000 ppm TREOnoCe for flat areas

\* Area Type can be sloping (sloping greater than 5 degrees) or flat. Sloping areas can potentially be mined by on site leaching, requiring lower costs.

\*\* Ampasibitika, Ampasibitika South and Caldera

\*\*\* Ampasibitika South and Caldera

TREO = LREO+HREO TREOnoCe = TREO-Ce<sub>2</sub>O<sub>3</sub>

HREO = Y<sub>2</sub>O<sub>3</sub>+Eu<sub>2</sub>O<sub>3</sub>+Gd<sub>2</sub>O<sub>3</sub>+Tb<sub>2</sub>O<sub>3</sub>+Dy<sub>2</sub>O<sub>3</sub>+Ho<sub>2</sub>O<sub>3</sub>+Er<sub>2</sub>O<sub>3</sub>+Tm<sub>2</sub>O<sub>3</sub>+Yb<sub>2</sub>O<sub>3</sub>+Lu<sub>2</sub>O<sub>3</sub>

LREO = La<sub>2</sub>O<sub>3</sub>+Ce<sub>2</sub>O<sub>3</sub>+Pr<sub>2</sub>O<sub>3</sub>+Nd<sub>2</sub>O<sub>3</sub>+Sm<sub>2</sub>O<sub>3</sub>

## 15 Market Studies and Contracts

No market study was undertaken for this study, however price assumptions from various REE technical report were compiled and spot prices were collected to establish the cut-off grade. A summary of the price assumption are shown in Table 14-8.

## 16 Environmental Studies, Permitting and Community Impact

### 16.1 Summary of Available Reports

SGS has consulted four (4) environmental monitoring reports (RSE: Rapport de Suivi Environnemental) covering the years 2009 to 2013 and an Annual Activity Report for 2013. These reports have been carried out 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 has 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) has 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 minimize 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 Tantalus for the 2009-2011 periods. However, the northeastern part of that exploration license, and an area of approximately 100 km<sup>2</sup>, 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 has 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 prioritized. 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 by Tantalus within its exploration license is not over yet, as the closure procedures have yet to be taken.

A draft contract between Tantalus and the INSTN has been done 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, extracting rare earths, and the transformation of that Tantalus plans to conduct. The study was set to begin in April of 2014. No confirmation of the status of this study was obtainable by the authors at the time of publication.

## 16.2 Permits

Tantalus holds a 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 also 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 carried 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) worth 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 by Tantalus 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-pit. Some modifications had to be made by Tantalus in order to respect this new activity. For selection of location for the 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.

### **16.3 Environment and Community Conclusions**

According to Tantalus, a baseline study will be finalized by end of 2014 and after that Tantalus will commence the social and environmental impact assessment which is expected to last end of 2015.

Madagascar has a very diverse and unique biological environment, and there are several groups (local and international) invested in protecting the habitat. Tantalus needs to continue to discuss 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.

In-situ leaching of REE in regolith is not currently used outside of China. Tantalus should educate the stakeholders on this method and plan to execute a small trial of this method to test its applicability and to this project but maybe more important allow stakeholders to witness the methodology at an early stage.

## 17 Adjacent Properties

There are no known mineral titles adjacent to the TRE project.

## 18 Other Relevant Data and Information

*This section was summarized from Gilbertson (2013) and has been updated to reflect new information.*

### 18.1 Country Profile

#### 18.1.1 Introduction

##### 18.1.1.1 Geography

The Republic of Madagascar is located approximately 420 km east of Mozambique off the coast of Africa in the Indian Ocean. It is the world's fourth largest island and covers an area of approximately 590,000 km<sup>2</sup>, extending over 1600 km from north to south and some 800 km east to west. Its population is approximately 21 million people. The capital and largest city is Antananarivo, often abbreviated and referred to as Tana.

The indigenous inhabitants of Madagascar fall into two main groups: those of largely Malayo-Indonesian descent and those principally of African descent. There are also small numbers of expatriate French and Asians. The official languages are Malagasy (a language of Indonesian origin), French and English. Over 50% of the people follow traditional religious beliefs: about 40% are Christian (equally divided between Roman Catholics and Protestants) and 7% are Muslim.

The national currency is the Ariary (MGA) which was reintroduced to replace the colonial Malagasy Franc (MGF). The Ariary is on a fixed exchange with the Malagasy Franc at 1 MGA to 5 MGF. Prices are still commonly quoted in both currencies. Madagascar is one of the poorest countries in the world, with annual per capita income of approximately USD 260. A total of 70% of the population are classified in poverty with half of all children malnourished.

##### 18.1.1.2 Politics

Madagascar held its second presidential election under the 1992 Constitution in 1996, following the impeachment of then President Albert Zafy earlier that year. The election was accepted widely as free and fair and the winner, former Second Republic President Didier Ratsiraka, took office in February 1997. Post 1997 Ratsiraka and his party, the Association for the Rebirth of Madagascar (AREMA), consolidated power and greatly weakened the previously strong non-AREMA parties.

Although power remains formally divided between the President, his Prime Minister, the Cabinet, and a bicameral legislature (Senate and National Assembly), the 1998 revision of the Constitution significantly strengthened the presidency, weakened the National Assembly, and gave the President the power to name one-third of the Senators. Indirect Senate elections held in March 1997 were considered to be generally free and fair, with mayors and provincial councils electing two-thirds of the new Senators, nearly all from AREMA.

In December 2001 presidential elections were held. However the results were disputed and a winner was not named by year's end. Most of the institutions provided for in the revised Constitution, including autonomous provincial Governments, were established during the year but their organisation and funding were also uncertain at year's end.

After the end of the 2002 political crisis, the new President Marc Ravalomanana began many reform projects, forcefully advocating "rapid and durable development" and the launching of a battle against corruption. In December 2002, legislative elections gave his newly formed Tiako-I-Madagasikara (I Love Madagascar) Party a commanding majority in the National Assembly. In November 2003 municipal elections were conducted freely, returning a majority of supporters of the President, but also significant numbers of independent and regional opposition figures.

In 2004 as part of the new Government, the President replaced provincial governors with appointed PDS's (Presidents des Delegations Speciales). Subsequent legislation established a structure of 22 regions to decentralise administration. Financing and specific powers for the regional administrations remained to be clarified.

Despite being re-elected in 2006, Ravalomanana's Government was forcibly removed in March 2009 in a militarily-backed uprising lead by Andry Rajoelina. Rajoelina formed a High Transitional Authority of which he became the self-appointed Transitional Head of State. In November 2010, a referendum was held to update the constitution.

Following the 2009 uprising and due to ensuing political, social and economic problems, the South African Development Community (SADC), with the participation of the leaders of South Africa, Botswana and Zambia, has been engaged in the mediation process.

On the 17<sup>th</sup> September 2011 it introduced a roadmap comprising numerous incentives to improve Madagascar's democratic and humanitarian performance, to facilitate a political truce, and to reintroduce Madagascar back into the international community. The roadmap is recognised and has been well-received by the international community. One of the conditions stipulated in the roadmap was that an interim Government would be formed with Rajoelina as President, even though this was strongly contested by many opposition parties. The roadmap also gives the Head of State the power to appoint a Prime Minister and stipulates that every exiled person who opposed the Government is free to return to their home state. This means the likes of former President Ravalomanana is permitted to return to Madagascar and re-enter national politics as a member of the opposition. He returned to Madagascar in September 2014 as a private citizen. Madagascar held a successful presidential election in late 2013 where president Hery Rajaonarimampianina was elected. He has been in office since January 2014.

#### 18.1.1.3 *Security*

The State Secretary of the Ministry of Interior for Public Security and the national police, which are under the State Secretary, are responsible for law and order in urban areas. The Ministry of Armed Forces oversees the army, the air force, the navy, and the gendarmerie. The gendarmerie has primary responsibility for security except in major cities and is assisted in some areas by regular army units in operations against bandit gangs and cattle thieves. After a number of years of decline, the military force has stabilised at approximately 22,000 troops, including the gendarmerie.

Village-level law enforcement groups enforce local traditional laws called "dina," particularly in areas where the Government's presence is weak. There continues to be occasional reports that police, gendarmes and dina authorities have committed human rights abuses.

There has been some serious crimes committed against foreigners but these occurrences are rare. Except for occasional petty crime, foreigners are relatively safe in Madagascar.

#### 18.1.1.4 *Economy*

The economy relies heavily on agriculture and fisheries. Shrimp is the leading export. Agricultural exports grew 5.2% with vanilla, coffee, cloves and pepper registering increases. Textiles were another major export. The smuggling of vanilla, gold and precious stones and cattle rustling continue to be major concerns. Overall economic performance has improved in recent years, but around 50% of the population remain in poverty and foreign assistance remains a major source of national income.

After discarding socialist economic policies in the mid-1990s, Madagascar has followed a World Bank- and IMF-led policy of privatisation and liberalisation. This strategy placed the country on a slow and steady growth path from an extremely low level. Agriculture, including fishing and forestry, is a mainstay of the economy, accounting for more than one quarter of GDP and employing 80% of the population. Exports of apparel have boomed in recent years primarily due to duty-free access to the US. However Madagascar's failure to comply with the requirements of the African Growth and Opportunity Act (AGOA) led to the termination of the country's duty-free access in January 2010.

Deforestation and erosion, aggravated by the use of firewood as the primary source of fuel, are serious concerns.

Former President Ravalomanana worked aggressively to revive the economy following the 2002 political crisis, which triggered a 12% drop in GDP that year. The current political crisis which began in early 2009 has dealt additional blows to the economy. Tourism dropped more than 50% in 2009, compared with the previous year. The real growth rate was 7% in 2008, but fell to minus 1% in 2009, attributed to the political crisis that occurred earlier in the year.

In 2011, the overall economy grew an estimated 0.6%, only slightly more than it did in 2010 (0.5%). It was mainly driven by the secondary (up 2.7% from 2010) and tertiary (up 2.1% from 2010) sector industries. The primary sector shrank by 2.3% because of poor agricultural output resulting from insufficient rainfall and several hurricanes.

Mining remained one of the economy's principal strengths and extractive industries grew an impressive 25.9%. The secondary sector's best performers were beverages, paper and food-processing and in the tertiary sector banking, telecommunications and transport, supported by tourism, which recovered in 2011 with a 14.8 % rise in visitors (from 196,052 compared to 225,055 in 2010).

Overall investment fell to 14.9% of GDP in 2011 from 18.8% in 2010 as a result of less development aid and the end of the building and installation phases of several large mining projects. In real terms, the drop was 11.2% and also affected public and private investment (down 8% and 12% respectively). Total consumption by volume was slightly up (1.1%) but private consumption rose slightly more (1.2%) than public (0.7%). Total consumption was 93% of GDP, down 2.7 percentage points from 2010, mainly because of private consumption dropping from 86.3% of GDP to 83.6%. Public consumption was steady at 9.5% of GDP (compared to 9.4% in 2010).

A combination of the roadmap agreement and increase activities in the mining sector are anticipated to increase growth in 2014 and 2015. Foreign aid (which funded 70% of Government investment) partially resumed in 2012. With the successful elections of 2013, more aid could return in 2015 and with it an improved business climate for the private sector.

It is anticipated that mining will be the chief engine of growth in 2014 and 2015, with production starting at Madagascar's largest mine at Ambatovy. The tertiary sector should benefit from revived tourism. Agricultural production will remain modest in 2014 and 2015 because of low rainfall and frequent hurricanes.

## 18.2 Mining Industry in Madagascar

Madagascar has good potential to discover and develop new deposits for a range of commodities, although its mining industry is underdeveloped. 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. The island has other large deposits containing gold, nickel-cobalt, heavy mineral sands (titanium), bauxite, copper, lead, manganese, platinum, zinc, zirconium, coal and petroleum products.

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.

According to the most recent United States Geological Survey (USGS) Minerals Yearbook for Madagascar (USGS, 2014), in 2012 Madagascar accounted for about 8% of world ilmenite production. The country was also one of the world's top-ranked sapphire producers in early 2008. Other domestically significant minerals produced included chromite and ornamental stones.

Despite the recent political situation there are still a sizeable number of foreign exploration and mining companies that have a presence in Madagascar. In 2010 (the date of the latest available figures), the mining sector grew by an estimated 121% (USGS, 2012). Furthermore, the companies operating in Madagascar appear to be involved with a variety of commodities, a testimony to the diversity of the country's perceived mineral wealth. Madagascar is also a recognised gold producer with major artisanal activity in various parts of the country.

### 18.2.1 Mining and Exploration Companies

Mining companies include Rio Tinto, who commenced exploitation at the QMM minerals sands project in southeast Madagascar in 2009. The project is located near to the coastal town of Fort Dauphin and has a production target of 2.2 Mt of ilmenite per year. Rio Tinto owns 80% of the operation and the Government of Madagascar owns the rest ([www.riotintomadagascar.com](http://www.riotintomadagascar.com)).

The Canadian company Sherritt International is developing a nickel and cobalt operation at Ambatovy, approximately 80 km east of Antananarivo. It is professed to be one of the world's largest nickel mines, tentatively producing 60,000 t of nickel per year and with a mine life of 27 years. The mine commenced operation in July 2010. Annual design capacity is for 60,000 t of nickel and 5,600 t of cobalt. Sherritt owns 40% of project with Japan's Sumitomo Corp and South Korea's state-run Korea Resources with 27.5%, and Canada's SNC Lavalin Group with 5% ([www.sherritt.mg](http://www.sherritt.mg)).

The South African company Exxaro completed a pre-feasibility study in 2009 and confirmed a large reserve of smelter-grade ilmenite between Toliara and Marombe in southwest Madagascar. A bankable feasibility study for the deposit is still underway ([www.exxaro.com](http://www.exxaro.com)).

The Australian company Red Island Minerals delineated a 180 Mt coal resource in the Sakoa area, southwest Madagascar. The project was recently acquired by the Thai company Petroleum Authority of Thailand. Mine development is underway leading to production in 2014 ([www.pttplc.com](http://www.pttplc.com)).

The Australian company Lemur Resources is a thermal coal exploration and development company with assets in Madagascar. Lemur is the 100% shareholder of Coal of Madagascar Limited which is the 99% Shareholder of the Malagasy registered company, Coal Mining Madagascar SARL ("CMM"). CMM has an interest in seven mining permits located in the Imaloto Coal Basin and work is currently focussed on the Imaloto coal project with an Inferred JORC resource of 176.6 Mt ([www.lemurresources.com](http://www.lemurresources.com)).

The Canadian company Energizer Resources (Energizer) is actively developing its Molo graphite project in southern Madagascar as part of a joint venture with Australian company Malagasy Minerals. Energizer has a 75% ownership interest and is the operator of the project. It recently completed its resource drilling programme consisting of 47 holes, totalling 9246 m. Energizer has recently released an updated NI 43-101 compliant resource and is waiting on a Preliminary Economic Assessment study ([www.energizerresources.com](http://www.energizerresources.com) and [www.malagasyminerals.com](http://www.malagasyminerals.com)).

The Australian company Aziana has been actively exploring for gold and bauxite since 2006 in various properties throughout Madagascar ([www.aziana.com.au](http://www.aziana.com.au)).

The UK company Jubilee Platinum is exploring for platinum approximately 160 km southwest of Antananarivo. Exploration is focussed on a layered mafic-ultramafic intrusive and the company is planning to advance the project with additional drilling ([www.jubileeplatinum.com](http://www.jubileeplatinum.com)).

The Canadian company Majescor Resources was actively exploring the historical Besakoa volcanogenic massive sulphide (VMS) deposit in south-western Madagascar ([www.sunridgegold.com](http://www.sunridgegold.com)).

Other companies exploring in Madagascar include:

- Austral Resources - zircon, western Madagascar ([www.austral-resources.com](http://www.austral-resources.com));
- Cline Mining - iron, eastern Madagascar ([www.clinemining.com](http://www.clinemining.com));
- Diamond Fields International - lateritic nickel, central Madagascar ([www.diamondfields.com](http://www.diamondfields.com));
- L P Hill - uranium and thorium, southern Madagascar ([www.lphill.com.au](http://www.lphill.com.au));
- Prom Resources - gold, central and south-western Madagascar ([www.promresources.com](http://www.promresources.com));
- UMC Energy - uranium, western Madagascar ([www.umc-energy.com](http://www.umc-energy.com)); and
- Zamarat Mining - gold, Madagascar ([www.zamaratmining.com](http://www.zamaratmining.com)).

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.

## 19 Interpretation and Conclusions

This report represents the first update to the mineral resource first published by SRK in 2013. The resource base has increased significantly since the maiden estimation in large part due to the pitting completed in the North West Territory. This area appears to have significant potential as a producing region, although metallurgical testing is required to validate that the REE are hosted in an “ionic” manner. This report also provides the first resources in the measured and indicated resources and the first application of a cut-off grade to resources in the ionic clays for the TRE project.

### 19.1 Metallurgical processing

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 TRE 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. An in-situ leaching trial should be planned with some form of participation from the various stakeholders so that the optimal processing methodology can be developed (best recovery with limited risk to the environment). Also 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 TRE project.

### 19.2 Environmental issues

The environmental impact to date is largely limited to activities associated with exploration activities. With a baseline study presently being executed, Tantalus will be in a better position to start a comprehensive Environmental and Social Impact Study (ESIS). The development of the TRE 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 recognized for its unique biodiversity.

### 19.3 Hydrology

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

## 19.4 Economic potential

The main impact relating to the economic potential of the project concerns the fluctuation of the sale prices for REE. Although cost of energy, cost of chemicals for processing, recovery of the metals have an impact on the economy of this type of project, their impacts compared to the risk to the REO sale price are minor to the overall economic viability of the project. Although prices have somewhat stabilized since the peak in 2011 (particularly the HREE) there are many factors in play. These include new REE production coming on-stream, quotas imposed by the Chinese government and even a crackdown on illegal mining in that country.

## 19.5 Ionic Clays REE Deposits

The TRE project is one of the only 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. One recent discovery of REE within ionic clays is the Mineração Serra Verde project within the municipality of Minaçu, Goiás State, Brazil. Mineração Serra Verde (MCV), a privately held project developer is studying this deposit that appears similar in size and grade. An April 2013 resource estimate included 412-million tonnes grading 0.16% at a 0.10% total rare earth oxide (TREO) cutoff grade (Lazenby, 2014). This resource is supposed to be according to NI 43-101; however the report is not available and little is known about the project including expected recoveries and HREE to LREE ratios.

## 20 Recommendations

Work to date on the TRE project has focused on outlining resources; this goal has largely been completed with a significant mineral inventory that can be incorporated in economic studies. For the next phase of work the authors would suggest a campaign of test mining. This is in line with the plans that the company has communicated. Many studies should be conducted in parallel at this stage to mitigate the various risks known for this project outlined below. These studies should culminate in a Feasibility Study which will provide further detail to the economic potential for this project.

### 20.1 Additional Metallurgical Test Work

Further to the preliminary metallurgical testwork done at Outotec, the University of Toronto and at SGS Lakefield, additional testwork should be planned to optimise the recovery of REE from the regolith material. It is recommended to set up a trial in-situ leaching program (test mining) with inputs from all stakeholders to test the optimal methods of extraction that limit environmental impact. This will illustrate the amenability of the material to in-situ 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.

### 20.2 LIDAR survey

Tantalus should select an external contractor to complete an airborne LIDAR (Light Detection and Ranging) survey over the project area. This will provide very accurate topographical data and include the collection of high spatial resolution ortho-imagery. These data is fundamental for the creation of a Digital Terrain Model (DTM) that will be used for a variety of applications including regolith / geomorphological studies, mineral resource estimation and environmental assessment.

### 20.3 Infrastructure Development

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 Tantalus should continue to make prepare infrastructure for the planned pilot mining.

### 20.4 Social and Environmental Programs

Tantalus will continue its social and environmental programmes 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 will be revisited as soon as more immediate requirements are met.

Tantalus has an environmental plan in place and dedicated personnel to execute it. However, the completion of a comprehensive environmental impact study and a formal environmental management plan remain requirements for the project due to the stringent environmental regulations set by the government of Madagascar.

Tantalus is in the final stages of completing an environmental and social baseline study. Once completed and all the other required input factors are ready, Tantalus should proceed with the environmental impact assessment required for the environmental permit as well as for the exploitation license. These activities are of utmost importance as no mining activity will be permitted without prior approval by the relevant authorities, as per the regulations on environmental protection and the commitments contained in the environmental impact study.

## 20.5 Proposed Budget for Recommended Work

The work described above has been broken down into a simplified budget in Table 20-1.

Table 20-1: Proposed budget for work to be completed in the next phase of work.

<b>Proposed Work</b>	<b>Estimated Cost (USD)</b>
LIDAR survey	200,000
Sampling and Testwork	400,000
Hydrological study	400,000
Surface Rights studies	500,000
Social and Environmental Studies	1,250,000
Pilot Mine preparations	2,000,000
Road Construction	2,000,000
Feasibility Study	1,000,000
<b>Total Studies Expenditure</b>	<b>7,750,000</b>
Contingency (20%)	1,550,000
<b>Total Budget</b>	<b>9,300,000</b>

## 21 References

Bao, Z. and Zhao, Z. 2008. Geochemistry of mineralization with exchangeable REY in the weathering crusts of granitic rocks in South China. *Ore Geology Reviews*. Vol. 33. pp. 519-535. (Bao+Zao-2008.pdf)

BGS. 2010. Rare Earth Elements. British Geological Survey. 45 p. (BGS-2010.pdf)

Chi, R. and Tian, J. 2008. Weathered crust elution-deposited rare earth ores. Nova Science Publishers Inc. 300 p. (Hardcopy only)

de Wit, M. J. 2003. Madagascar: Heads it's a continent, tails it's an island. *Annu. Rev. Earth Sci.* Vol 31. pp. 213-248.

<http://www.chinafolio.com/china-rare-earths-and-technological-edge/>

DOE (U.S. Department of Energy). Dec 2011. "Critical Materials Strategy." Washington, DC: DOE.

Earthmaps Consulting. 2009. Tantalus REE-Ta-Nb-Zr-(Hf-U-Sn) project Madagascar - Interpretation of helicopter magnetic and radiometric data. Report volume 1/1. 36 p.

Estrade, G., Salvi, S., Beziat, D., Boix, M. and Soatsitohaina, R. 2011a. HFSE enrichment in a peralkaline granite related skarn, Ampasindava peninsula, Madagascar. 36 p. (Peralkaline granite related skarn - Ampasibitika intrusion.pdf)

Estrade, G., Salvi, S., Beziat, D. 2011b. Rare metal mineralization associated to Cenozoic peralkaline magmatism, Ampasindava peninsula, Madagascar. University of Toulouse. 1p. (Abstrcat.docx)

Estrade, G., Salvi, S., Beziat, D. 2012. Nouveau gisement de métaux rares associé au complexe alcalin oligo-miocène d'Ampasindava, Madagascar. University of Toulouse. 1 p. (120312\_Estrade Guillaume\_Abstract De Launay ss.docx)

Fugro Consult GMGH. 2008. The Tantalus Tantalum-Niobium and rare earth project: Interim report on new data plus an exploration programme 2008 - 2010 with budget estimate. 28 p.

Ganzeev, A. A. and Grechishchev, O. K. 2003. A new genetic type of rare-metal alkali granites of Madagascar. *Russian Geology and Geophysics*. Vol. 44. No. 6. pp. 539-553. (ampasindava\_alkali\_granites.pdf)

Gilbertson, J. 2013, A Competent Persons Report on the Tantalus Project, Northern Madagascar, Prepared for Tantalus Rare Earths AG, Competent Persons Report, SRK Exploration Services Ltd. 146 Pages. January 2013.

Lacroix, A. 1922. *Minéralogie de Madagascar. Tome I - Géologie-Minéralogie descriptive*. Challamel, A. (ed.). Paris. pp. 589-590.

Lazenby, H. 2014. MSV aims to lead Western rare earths production with low-cost project: <http://www.miningweekly.com/article/msv-aims-to-lead-western-rare-earths-production-with-low-cost-project-2014-03-11>

Levinson, A. A. 1974. Introduction to exploration geochemistry. Applied Publishing Calgary. 612 p.

Moldoveanu, G. and Papanagelakis, V. G. 2012a. Recovery of rare earth elements adsorbed on clay minerals: I. Desorption mechanism. Hydrometallurgy. Vol. 117-118. pp. 71-78. (Recovery of rare earth elements adsorbed on clay minerals- I. Desorption mechanism2.pdf)

Moldoveanu, G. and Papanagelakis, V. G. 2013a. Recovery of rare earth elements adsorbed on clay minerals: II. Leaching with ammonium sulphate. Hydrometallurgy. Vol. 131-132. pp. 158-166.

Moldoveanu, G. and Papanagelakis, V. G. 2013b. Leaching of rare earths elements from clay materials. Summary report to Tantalus Rare Earths AG. Department of Chemical Engineering and Applied Geochemistry, University of Toronto. 9 p.

Melluso, L., Morra, V., Brotzu, P., Franciosi, L., Grifa, C., Lustrino, M., Morbidelli, P., Riziky, H. and Vincent, M. 2007. The Cenozoic alkaline magmatism in central-northern Madagascar: a brief overview. Pers. Mineral. Vol. 76. pp. 169-180. (2007 Alkaline Magmatism Madagascar.pdf)

Nesbitt, H. W. 1979. Mobility and fractionation of rare earth elements during weathering of a granodiorite. Nature. Vol. 279. pp. 206-210.

OMNIS-SM. 1992a. Carte geologique du complexe volcano-plutonique d'Ambohimirahavavy aux elements de metallogenie. 1: 25 000. 1 p.

OMNIS-SM. 1992b. Exploration results. Evaluation of the rare metals deposit of Ampasibitika (1990 field season). 16 p.

OMNIS-SM. 1992c. Exploration results of prospecting-evaluation of the rare metals deposit of Ampasibitika (1990 field season). Contract No. 75 - 517 / 10200. 125 p.

OMNIS-SM. 1992d. Projet metaux rares resultats de analyses spectrales et chimiques. Contrat No. 75 - 517 / 10200. 206 p.

OMNIS-SM. 1992e. Projet metaux rares resultats de analyses spectrales et chimiques - Analyses. 1987/1992. 38 p.

OMNIS-SM. 1990. Resultats des travaux de prospection-evaluation du gisement de metaux rares D'Ampasibitika. Contrat No. 75 - 517 / 10200. Campagne 1990. 100 p.

OMNIS-SM. 1989. Resultats des travaux de recherches realises en 1989 dans la province D'Ampasindava. Contrat No. 75 - 517 / 10200. 100 p.

Orris, G. J. and Grauch, R. I. 2002. Rare earth element mines, deposits and occurrences. United States Geological Survey. Open-file report 02-189. 174 p. (Orris+Grauch-2002.pdf)

SRK ES. 2011a. An independent Competent Person's Report on the Exploration assets of Tantalus Rare Earths AG. 81 p.

SRK ES. 2011b. Addendum to January 2011 independent Competent Persons Report on the Tantalus project, Madagascar. 9 p.

Tantalus Rare Earths. 2012a. Regolith logging and sampling notes. Tantalus Rare Earths project, Madagascar. 9 p.

Tantalus Rare Earths. 2012b. Lateritic REE ore. 12 p.

USGS. 2012. 2010. Minerals Yearbook - Madagascar (advance release). US Department of the Interior. By Yager, T. R. United States Geological Survey. 6 p. (USGS-2012.pdf)

USGS. 2011. Carbonatite and alkaline intrusion-related rare earth element deposits - a deposit model. Open-file report 2011-1256. United States Geological Survey. 8 p. (2011\_USGS\_Carbonatite alk intrusion related REE mineralisation.pdf)

Zuoping, Z. and Chuanxian, L. 1996. The behavior of rare-earth elements (REE) during weathering of granites in Southern Guangxi, China. Chinese Journal of Geochemistry. Vol. 15. No. 4. 9 p. (REE-clays-China.pdf)

## Appendix A. Certificate of Qualified Person Guy Desharnais

To accompany the Report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20th, 2014.

I, Guy Desharnais, of Laval Quebec, do hereby certify that:

1. I currently work as a geologist at SGS Canada Inc. - Geostat with an office located at 10 boul. de La Seigneurie Est, Suite 203, Blainville, Quebec, Canada J7C 3V5;
2. This certificate regards the technical report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20<sup>th</sup>, 2014;
3. I have a B.Sc. honours degree from the geology department of the University of Manitoba and I received a Ph.D. in 2004 from the same university. I have worked on several technical reports regarding resource estimations and exploration of REE and laterite hosted deposits. I am a registered member of the Ordre des Géologues du Québec (#1141). I am a ‘Qualified Person’ for the purposes of National Instrument 43-101 (the ‘Instrument’);
4. I visited the Tantalus Project between November 28<sup>th</sup> and December 8<sup>th</sup> 2013;
5. I am responsible for sections 15 to 20 of this report;
6. I am independent of Tantalus as described in section 1.5 of the Instrument;
7. I have had no prior involvement with the property that is the subject of this report;
8. I have read the Instrument and the sections of this report that I am responsible for have been prepared in compliance with the Instrument; and
9. As of the effective date of this report, to the best of my knowledge, information, and belief, the parts of this report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the report not misleading.

Signed, sealed and dated November 27<sup>th</sup> at Blainville, Quebec.

(Original copy signed and sealed)

**Guy Desharnais, Ph.D., P.Geo.** Geological Services Technical Manager

SGS Canada Inc. - Geostat

## Appendix B. Certificate of Qualified Person Yann Camus

To accompany the Report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20th, 2014.

I, Yann Camus, do hereby certify that:

1. I am project engineer with SGS Canada Inc. - Geostat with an office at 10 Blvd Seigneurie East, Suite 203, Blainville, Quebec, Canada, J7C 3V5;
2. This certificate regards the technical report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20<sup>th</sup>, 2014;
3. I am a graduate from École Polytechnique de Montréal in 2000. I am an engineer and a registered member of the Ordre des Ingénieurs du Québec (#125443). I have worked as a geological engineer continuously since my graduation from university. My technical expertise includes resources evaluation. I have been involved in several resource, pre-feasibility and feasibility studies as well as preliminary economic assessments. I have participated in worldwide projects in rare earths, rare and base metals, iron, bauxite and industrial minerals;
4. I have not visited the Property;
5. I am responsible for section 14 of this report;
6. I am independent of Tantalus as described in section 1.5 of the Instrument.
7. I have had no prior involvement with the property that is the subject of this report;
8. I have read the Instrument and the sections of this report that I am responsible for have been prepared in compliance with the Instrument; and
9. As of the effective date of this report, to the best of my knowledge, information, and belief, the parts of this report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the report not misleading.

Signed, sealed and dated November 27<sup>th</sup> at Blainville, Quebec.

(Original copy signed and sealed)

**Yann Camus, Eng.** Project Engineer

SGS Canada Inc. – Geostat

## Appendix C. Certificate of Qualified Person Claude Bisailon

To accompany the Report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20th, 2014.

I, Claude Bisailon, of Pointe-Claire, Quebec, do hereby certify that:

1. I currently work as an engineer at SGS Canada Inc. with an office located at 10 boul. De la Seigneurie Est, Suite 203, Blainville, Québec, Canada J7C 3V5;
2. This certificate regards the technical report entitled “NI 43-101 Technical Report - Resources for the Tantalus Rare Earth Ionic Clay Project, Northern Madagascar” with an effective date of October 20<sup>th</sup>, 2014;
3. I have a B.Sc. in geology from Concordia University in Montréal and I received B.Eng. in Geological Engineering from Université Laval in Quebec city in 1996. I have worked on several technical reports regarding resource estimations and exploration. I am a registered member of the Ordre des Ingénieurs du Québec (#116407). I am a ‘Qualified Person’ for the purposes of National Instrument 43-101 (the ‘Instrument’);
4. I visited the Project on April 26th to May 3rd and September 12th to 28th, 2013;
5. I am responsible for sections 1 to 13 of this report;
6. I am independent of Tantalus as described in section 1.5 of the Instrument;
7. I have had no prior involvement with the property that is the subject of this report;
8. I have read the Instrument and the sections of this report that I am responsible for have been prepared in compliance with the Instrument; and
9. As of the effective date of this report, to the best of my knowledge, information, and belief, the parts of this report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the report not misleading.

Signed, sealed and dated November 27<sup>th</sup> at Blainville, Quebec.

(Original copy signed and sealed)

**Claude Bisailon, Eng.** Project Engineer

SGS Canada Inc. - Geostat