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TRANS-TASMAN RESOURCES LIMITED

South Taranaki Ironsand Project Resource Report

Submitted to:
Matt Brown
GM Exploration
Trans-Tasman Resources Limited
Level 13, 342 Lambton Quay
PO Box 10-571
Wellington 6143
NEW ZEALAND

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REPORT

Golder Associates

Executive Summary

This report details the mineral resource update for the South Taranaki titanomagnetite deposit located offshore southwest of the North Island of New Zealand. The resource estimates are based on all available assay data as of 4 September 2013.

The deposit is a submarine aeolian/alluvial/marine accumulation of ironsand in palaeo channels, strandlines and dunes. Geophysical surveys have identified strongly magnetic targets throughout the exploration area. The drilling to date has penetrated what is interpreted as a blanket of overburden sand covering the geophysical anomalies. Several deeper holes have defined the depth of sand as being up to 30 m.

The resource has been estimated using an Ordinary Kriging algorithm. The screened recovery (REC) has been applied to the models. Head grades and tonnages are for all material less than 2 mm in diameter. Davis Tube Concentrate (DTC) grades represent the magnetically recoverable portion of the sample. Head grades are estimated using samples weighted by REC. Estimations for concentrate grades are weighted by REC and Davis Tube Recovery (DTR). The weighting has been applied in order to appropriately reflect the relationship between the REC and head assays for the head samples and REC, DTR and the DTC assays for the magnetic concentrate samples. Weighting has been completed by calculating the accumulation (REC × Head Grade, Rec × DTR × DTC assay) and subsequently back calculating the head and DTC assay estimates by dividing by relevant estimated REC and DTR values. Concentrate tonnage is calculated from the head tonnage and DTR.

Head grades have been estimated for Fe_2O_3, Al_2O_3, P_2O_5, SiO_2, TiO_2, CaO, K_2O, MgO, MnO, LOI, Recovery and DTR. The DTR is a laboratory scale representation of the potential metallurgical recovery of the processing plant. DTC grades have been estimated for Fe, Al_2O_3, P, SiO_2, Ti, CaO, K_2O, MgO, Mn, and LOI. DTR and DTC values are confined to 1289 samples in the proposed mining area.

The resource estimates are classified in accordance with the Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2012) as Indicated and Inferred. The recoverable Mineral Resource is reported from the block models *Area2_25_09_2013a.bmf (domains 1-7 and 9)* and *south_acc_24_11_2012.bmf (domain 8)*. No new data has been acquired for the Koitiata area therefore the model *south_acc_24_11_2012.bmf* was not updated in 2013.

The models have been reported at 3.5% DTR cut-off grade where DTR analyses were available within the proposed mining area (Table A and Table B). Outside this area a cut-off grade of 7.5% Fe_2O_3 is used based on the statistical relationship between Fe_2O_3 and DTR (Table C).

For comparison with previous resource estimates Table D reports the models at 5% Fe_2O_3 (head) cut-off grade.

Table A: Tonnage and Head Grades (%) – Proposed Mine Area – 3.5% DTR Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	DTR*	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	1	165.4	11.31	7.37	11.07	53.57	1.15	10.64	1.12	5.41	0.20	0.22	2.68	94.55
	3	480.1	11.64	7.57	12.70	51.38	1.19	10.98	1.15	5.36	0.21	0.26	2.24	98.24
	6	304.0	9.71	5.88	13.11	53.15	1.00	11.01	1.18	4.80	0.18	0.24	2.65	96.06
	7	81.6	10.52	6.08	10.87	49.67	1.03	13.94	1.00	5.92	0.20	0.23	4.23	88.00
	9	3.9	8.26	4.66	14.16	53.64	0.82	11.04	1.23	4.48	0.17	0.23	2.59	98.38
Indicated Total		***1035.1***	***10.92***	***6.91***	***12.42***	***52.13***	***1.11***	***11.17***	***1.14***	***5.24***	***0.20***	***0.24***	***2.59***	***96.20***
Inferred	1	26.9	15.85	11.82	9.12	49.60	1.60	9.61	0.91	5.43	0.23	0.19	5.16	92.90
	6	5.2	10.07	5.79	13.37	51.47	1.04	11.67	1.13	5.22	0.20	0.26	2.32	95.12
	7	3.4	12.52	8.12	9.64	48.89	1.21	13.82	0.82	6.82	0.21	0.20	3.90	90.84
Inferred Total		***35.5***	***14.68***	***10.58***	***9.79***	***49.80***	***1.48***	***10.32***	***0.94***	***5.53***	***0.22***	***0.20***	***4.63***	***93.03***
Total		**1070.7**	**11.04**	**7.03**	**12.33**	**52.05**	**1.12**	**11.14**	**1.14**	**5.25**	**0.20**	**0.24**	**2.66**	**96.10**

*the DTR estimate is based on analytical DTR and calculated DTR values.

Table B: Tonnage and Concentrate Grades (%) – Proposed Mine Area – 3.5% DTR Cut-Off Grade

Class	Domain	Mt	Fe	Al_2O_3	SiO_2	Ti	CaO	K_2O	MgO	Mn	P	LOI
Indicated	1	12.2	57.43	3.66	3.50	5.01	0.95	0.09	3.22	0.51	0.10	-3.17
	3	36.3	56.58	3.67	4.18	5.09	1.06	0.12	3.26	0.23	0.11	-3.04
	6	17.9	56.62	3.70	4.29	5.05	1.08	0.12	3.25	0.51	0.10	-3.07
	7	5.0	56.79	3.77	4.05	4.97	1.10	0.10	3.33	0.51	0.10	-3.12
	9	0.2	55.26	3.75	5.71	5.03	1.32	0.17	3.38	0.50	0.12	-2.93
Indicated Total		***71.5***	***56.73***	***3.68***	***4.10***	***5.06***	***1.05***	***0.12***	***3.26***	***0.37***	***0.10***	***-3.08***
Inferred	1	3.2	59.48	3.55	1.62	4.87	0.53	0.03	2.98	0.52	0.07	-3.38
	6	0.3	56.00	3.76	4.98	5.04	1.24	0.14	3.34	0.51	0.11	-3.07
	7	0.3	58.53	3.67	2.48	4.85	0.77	0.05	3.17	0.51	0.07	-3.27
Inferred Total		***3.8***	***59.06***	***3.58***	***2.02***	***4.88***	***0.62***	***0.05***	***3.03***	***0.52***	***0.08***	***-3.34***
Total		**75.3**	**56.83**	**3.67**	**4.02**	**5.06**	**1.03**	**0.11**	**3.25**	**0.37**	**0.10**	**-3.09**

*the DTR estimate is based on analytical DTR and calculated DTR values.

Table C: Tonnage and Head Grades (%) – Outside Proposed Mine Area – 7.5% Fe_2O_3 Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	2	119.9	8.68	13.05	49.57	0.88	13.88	1.17	5.26	0.19	0.25	4.19	84.64
	3	56.3	9.23	14.02	51.32	0.93	12.14	1.19	5.25	0.19	0.26	2.50	92.76
	4	71.3	9.94	11.80	46.23	0.95	16.21	0.90	6.22	0.21	0.26	5.02	88.12
	5	37.3	9.11	14.17	50.43	0.91	12.68	1.21	5.65	0.20	0.27	2.31	84.36
	6	100.6	11.28	13.09	51.87	1.18	10.74	1.13	4.66	0.20	0.22	2.83	94.05
	7	282.1	8.92	13.73	51.09	0.89	12.61	1.21	5.34	0.20	0.24	2.73	89.14
	9	123.7	9.07	14.14	51.53	0.90	12.18	1.20	5.33	0.19	0.26	2.25	93.06
Indicated Total		***791.2***	***9.33***	***13.48***	***50.57***	***0.94***	***12.79***	***1.16***	***5.33***	***0.20***	***0.25***	***3.06***	***89.64***
Inferred	1	33.2	15.18	7.88	47.61	1.52	13.35	0.79	5.58	0.21	0.21	5.87	88.42
	2	171.7	8.49	14.40	50.33	0.87	12.64	1.29	4.82	0.18	0.24	3.50	86.74
	3	108.5	8.89	14.68	52.31	0.91	11.04	1.33	4.63	0.18	0.25	2.55	94.68
	4	93.3	8.87	11.21	45.66	0.85	17.48	0.90	6.11	0.20	0.23	6.35	83.62
	5	4.2	8.42	13.51	50.23	0.82	13.77	1.16	6.20	0.20	0.28	2.62	79.70
	6	279.6	11.17	12.16	51.43	1.13	11.55	1.08	5.13	0.20	0.22	3.06	94.30
	7	144.6	8.67	11.11	45.19	0.84	17.70	0.91	5.88	0.20	0.23	6.87	82.45
	8	60.6	9.08	9.12	54.13	0.78	12.62	0.85	6.82	0.18	0.14	4.50	90.51
	9	190.8	8.95	14.41	51.59	0.89	12.22	1.23	5.54	0.19	0.27	1.90	90.78
Inferred Total		***1086.5***	***9.59***	***12.65***	***50.08***	***0.95***	***13.24***	***1.10***	***5.40***	***0.19***	***0.23***	***3.83***	***89.58***
Total		**1877.7**	**9.48**	**13.00**	**50.29**	**0.95**	**13.05**	**1.13**	**5.37**	**0.19**	**0.24**	**3.50**	**89.60**

Table D: Tonnage and Head Grades (%) – Full Area Reported – 5% Fe_2O_3 Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	1	213.5	10.25	11.18	54.86	1.04	10.44	1.16	5.08	0.18	0.20	2.92	93.45
	2	330.3	7.34	13.55	50.28	0.76	13.56	1.29	4.16	0.15	0.23	5.49	87.50
	3	617.8	10.74	13.20	52.20	1.10	10.70	1.22	5.04	0.19	0.25	2.32	97.75
	4	77.4	9.71	11.96	46.48	0.93	16.11	0.92	6.14	0.20	0.26	4.98	88.58
	5	110.8	7.27	14.87	52.35	0.77	11.48	1.42	3.91	0.15	0.24	4.05	90.65
	6	606.2	8.92	13.21	55.07	0.93	10.13	1.26	4.34	0.17	0.22	2.70	95.68
	7	569.8	8.36	14.07	52.39	0.85	11.76	1.31	4.72	0.18	0.23	2.95	90.24
	9	157.3	8.64	14.40	52.08	0.87	11.87	1.24	5.01	0.18	0.25	2.37	93.92
Indicated Total		***2683.0***	***9.07***	***13.37***	***52.70***	***0.93***	***11.39***	***1.25***	***4.69***	***0.18***	***0.23***	***3.13***	***93.30***
Inferred	1	64.0	14.93	8.54	48.83	1.50	11.76	0.87	5.37	0.21	0.20	5.73	90.00
	2	340.0	7.61	15.30	50.84	0.80	12.19	1.40	3.98	0.16	0.23	4.00	87.91
	3	179.9	8.11	15.37	53.85	0.85	10.05	1.46	4.04	0.16	0.24	2.44	95.79
	4	173.1	7.84	11.10	44.57	0.75	18.59	0.90	5.52	0.18	0.22	8.08	83.78
	5	7.5	7.57	13.45	52.20	0.77	12.61	1.28	5.19	0.17	0.26	3.34	83.64
	6	377.0	9.97	12.76	53.14	1.03	10.64	1.21	4.60	0.18	0.21	3.18	94.70
	7	315.7	7.52	12.48	48.00	0.75	15.72	1.12	4.76	0.17	0.22	6.43	85.56
	8	191.6	7.04	9.68	56.77	0.64	11.65	1.01	5.11	0.14	0.14	5.45	90.83
	9	529.7	7.38	16.21	52.95	0.76	11.10	1.45	4.17	0.16	0.25	2.18	91.57
Inferred Total		***2178.6***	***8.17***	***13.64***	***51.56***	***0.83***	***12.44***	***1.25***	***4.52***	***0.17***	***0.22***	***4.14***	***90.26***
Total		**4861.6**	**8.67**	**13.50**	**52.19**	**0.89**	**11.86**	**1.25**	**4.62**	**0.17**	**0.23**	**3.58**	**91.94**

Table of Contents

TABLES

FIGURES

APPENDICES

APPENDIX A
Licence/Permit Documents

APPENDIX B
Sampling Procedures

APPENDIX C
Bulk Density Determination

APPENDIX D
QAQC Plots

APPENDIX E
Drill Spacing Study

APPENDIX F
EDA Plots

APPENDIX G
Variography Plots

APPENDIX H
Block Model Validation Plots

APPENDIX I
Block Model Validation/Comparison

APPENDIX J
Limitations

1.0 INTRODUCTION

1.1 Preamble

1.1.1 Scope

Golder Associates Pty Ltd (Golder) was originally engaged by Trans-Tasman Resources Ltd (TTRL) to assist with the development of TTRL's South Taranaki ironsand project in New Zealand on 27 November 2009. This report details the latest update of the mineral resource and includes drilling results up to 4 September 2013.

1.1.2 Previous Work

In November 2009 an *in situ* maiden resource of 1040 Mt at 5.88% Fe was defined for the Taranaki South ironsand project . Golder (2009) report details the data analysis and geological interpretation supporting the resource. In July 2011, after additional drilling, the mineral resource was updated to 2121 Mt at 5.64% Fe (Golder, 2011).

This report details the results of the fifth update to the mineral resource. Table 1-1 summarises the various updates. For the 2012 work Davis Tube Recovery (DTR) and Davis Tube Concentrate (DTC) analyses became available for a subset of the resource in the proposed mine area.

Table 1-1 Mineral Resource History

Year	Month	Area	Mt	Head Fe%	Head DTR	Concentrate Mt	Concentrate Fe%
2009	November	Full	1040	5.88	-	-	-
2011	July	Full	2121	5.64	-	-	-
2012	January	Full	4638	6.23	-	-	-
2012	November	Full	4660	6.25	-	-	-
2012	November	Mine	1031	8.35	7.75	79.9	57.21

1.1.3 Golder Associates

Golder has been delivering quality technical solutions to the global mining sector for over 50 years, providing a comprehensive suite of integrated mining services, from concept study to mine closure. Golder has extensive practical experience in all aspects of design, planning and operation of open pit, underground and strip mines, enabling clients to realise the maximum value from mining projects. With 160 offices, our clients have access to pragmatic mine planning and mineral evaluation solutions, no matter where their mining project is located throughout the world.

Our integrated consulting, design and construction solutions operate at every stage of a mining project and are provided by teams specialising in:

- Resource and Reserve estimation
- Mine Planning and Ore Evaluation
- Integrated Tailings and Waste Management
- Rock Mechanics and Mine Geotechnical Engineering
- Mine Environment
- Mine Water
- Mine Infrastructure
- Heap Leach Design and Construction
- Mine Energy Optimisation and Carbon Management
- Sustainability and Corporate Social Responsibility
- Mine Safety and Industrial Hygiene
- Management Consulting.

1.2 Conventions

Grid References

Easting, Northing and RL references used in this report relate to WGS 84 Zone 60, unless specifically stated otherwise. All elevations are in metres.

Block Dimensions

Three-dimensional entities in this report are described in the format x by y by z, where x refers to the Easting distance in metres, y refers to the Northing distance in metres and z refers to the RL or vertical distance in metres.

Variograms

The experimental and model semi-variograms or correlograms produced as part of the geostatistical analysis are referred to as variograms.

Orientations

Linear orientations are defined in terms of plunge and plunge direction (i.e. plunge→plunge direction). Resultant planes are defined in terms of dip and dip direction (i.e. dip→dip direction).

In conventional three-dimensional variographic analysis, directional variograms are modelled in three principal directions as follows:

- Major axis (in the direction of the plunge of the mineralisation).
- Semi-major axis (perpendicular to the major axis but still in the plane of the mineralisation).
- Minor axis (in the direction across the dip of the ore body orthogonal to the other two axes).

These directions relate respectively to the strike, down dip and cross-strike directions only if the major axis is modelled in the same direction as the strike of the geological plane (i.e. at zero plunge). For this reason, the terms 'major axis', 'semi-major axis' and 'minor axis' are used to describe the principal directions.

Analytical Reporting

The South Taranaki resource is a titanomagnetite ironsand deposit. Titanomagnetite is $Fe^{2+}(Fe^{3+},Ti)_2O_4$, pure magnetite is Fe_3O_4. The analysis process reduces all compounds to oxides and reports these. For head samples standard analyses return iron results as Fe_2O_3 (Hematite), Fe is calculated from the stoichiometric ratios of Fe to O in the Fe_2O_3. The Fe content of a sample is the Fe_2O_3 value multiplied by 0.6994. For Davis Tube Concentrate sample analysis iron grades are reported as Fe.

Golder has estimated and reported the Fe_2O_3 content for the head grades and Fe for the concentrate grades of the deposit based on the analytical results.

For concentrate analyses negative loss on ignition (LOI) values are not uncommon. A negative LOI indicates a gain on ignition through oxidation of ferrous to ferric iron.

In historical documentation TTRL have reported TiFe. The TiFe ("Titanomagnetite") content of the deposit can be back calculated from the Fe_2O_3 content based on the assumptions and stoichiometric formula defined by TTRL and detailed in Section 4.5.

General Terminology

Abbreviations and terms used in this report are listed in Table 1-2.

Table 1-2: Abbreviations

Abbreviation	Meaning/Description
Accuracy	The ability to obtain the correct result
ASCII	American Standard Code for Information Interchange
ASX	Australian Stock Exchange
Blank	Sample without metal content to check possible contamination during assaying (e.g. crushed glass)
Cut-off	Grade above which mineralised material is considered to be ore.
CSV	Character Separated Variables, ASCII file
DTC	Davis Tube Concentrate
DTM	Digital terrain model – Electronic computer model of topography
DTR	Davis Tube Recovery
Duplicate	Sample that has been split from another to check the field sampling or laboratory's precision
GIS	Geographical Information System
GNS	Geological and Nuclear Sciences Limited

Abbreviation	Meaning/Description
Golder	Golder Associates Pty Ltd
HM	Heavy Mineral
ID^n	Inverse Distance ^{power}, e.g. ID^2, ID^3
JORC	Australasian Joint Ore Reserves Committee
Kriging	Grade estimation technique incorporating variability by distance
LOI	Loss On Ignition
ML	Mining Lease
NATA	National Association of Testing Authorities, Australia
NIWA	National Institute of Water and Atmospheric Research
NQ	Diamond drill Core Size (75.7 mm hole 47.6 mm core)
NZDS	New Zealand Dive and Salvage
OD	Outer Diameter
Ordinary Kriging (OK)	Estimation of grades into block model using a grade estimation technique incorporating variability by distance
Ore	Mineralised material that can be economically mined
ppb	Parts per billion
ppm	Parts per million – 10 000 ppm = 1%
Precision	The ability to obtain the same result each time
RC	Reverse Circulation
RES	Resource Evaluation Services
RL	Reduced Level. Commonly height above/below sea level
Standard Sample	Specially prepared sample whose metal grade is very accurately known and certified
SEM	Scanning Electron Microscope
TiFe	Titanomagnetite
TTRL	Trans-Tasman Resources LTD
Variogram	Mathematical and graphical way of representing variation of data as a function of separation distance for a given direction
Vulcan	Computer program by Maptek that is used to carry out resource estimation and mine planning – www.vulcan3D.com.
WASSP	Multi-beam sonar system www.wassp.com
XRF	X-Ray Fluorescence – analytical technique.

2.0 LOCATION AND ACCESS

TTRL hold exploration tenements to the north and south of Taranaki, off the west coast of the North Island of New Zealand. The mineral resources defined to date is located offshore from the Patea River mouth approximately 50 km north of Wanganui and 25 km south of Hawera (Figure 2-1).

The area is accessed by boat out of Wanganui harbour, a 180 km drive north of Wellington.

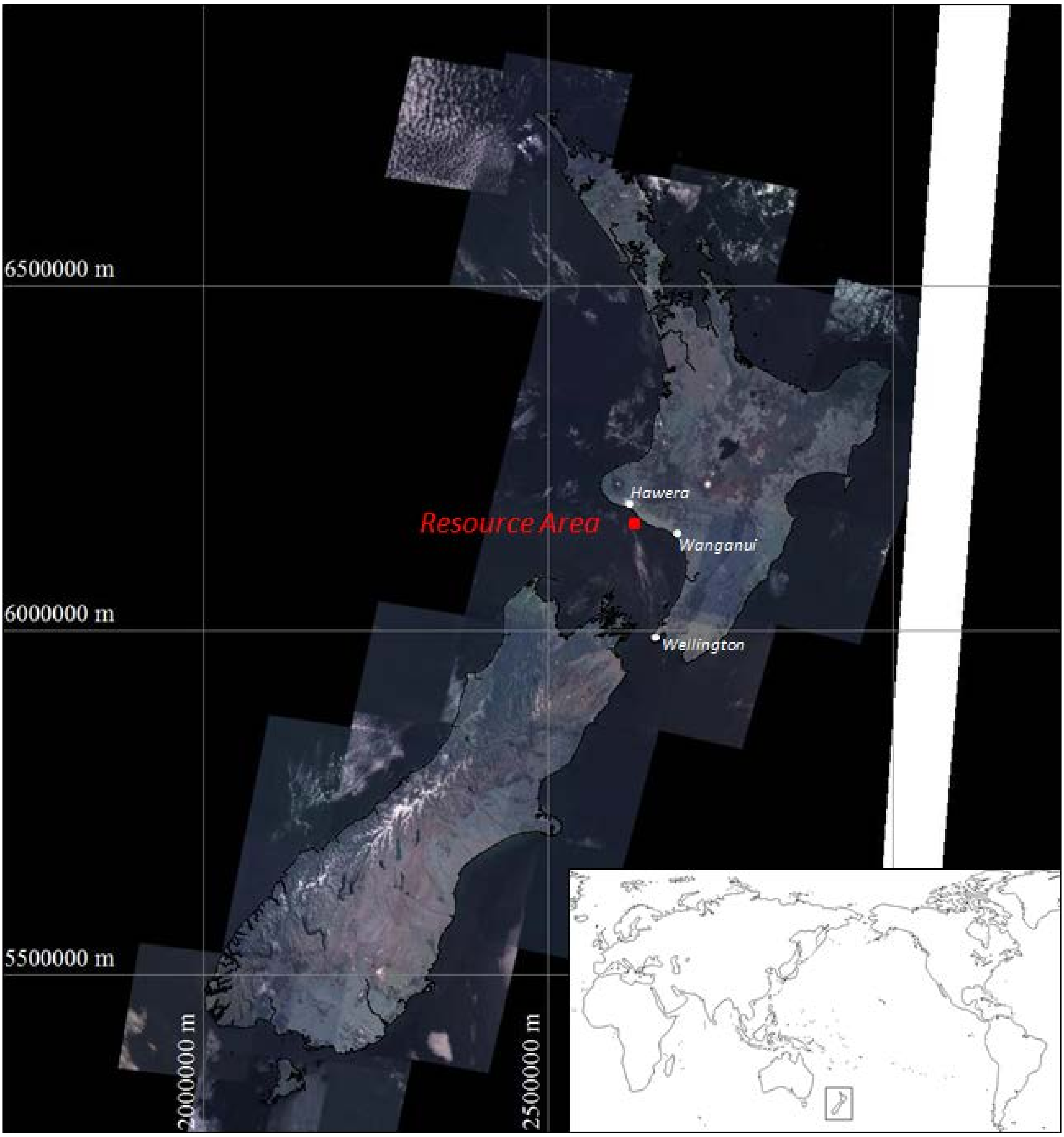


Figure 2-1: Location (LANDSAT Imagery)

2.1 Permits and Licences

The TTRL permits and licences are applicable to locations in New Zealand offshore to the north and south of Taranaki. TTRL has been granted Subsequent Exploration Permit (EP) 54068 which covers the northern half of what was Prospecting Permit (PP) 50383 off shore from Wanganui (Figure 2-2). The remainder of PP50383 is now covered by EP54270, EP54271 and EP54272 currently awaiting approval by the New Zealand Petroleum and Minerals division of the Ministry of Business, Innovation and Employment.

Exploration Permits allow for more detailed investigation of the tenements to be undertaken. They are only applicable within the 12 nautical mile limit around New Zealand. Beyond this limit the Continental Shelf Act applies and a different licence must be applied for. TTRL holds Continental Shelf Licence 50753 immediately south of EP54068 and EP54272 (Figure 2-2). Table 2-3 and Section 2.1.1 outline the permit and licence entitlements.

TTRL have submitted an application for a mining permit over the most prospective area in the exploration permits (MP55581 – Figure 2-3).

Table 2-1 lists the New Zealand Crown Minerals details held for each permit. Table 2-2 lists the details for the Continental Shelf Licence and Mining Permit. The full licence/permit documents for CSA50753 and EP54068 and Crown Minerals Permit summaries are reproduced in Appendix A. It is anticipated that the other permits will include the same conditions.

Note that Golder can only comment that appropriate permits are required and appear to be in place or are in progress for final approval. Golder does not offer any legal opinion about these tenements or their status.

Table 2-1: TTRL Exploration Permits

	No. 54068	No. 54270	No. 54271	No. 54272
Commodity:	MINERALS	MINERALS	MINERALS	MINERALS
Type:	Exploration Permit	Exploration Permit	Exploration Permit	Exploration Permit
Owners:	TTRL	TTRL	TTRL	TTRL
Location:	Taranaki	Waikato	Waikato	Taranaki
Operation Name:	-	Waikato North	Taharoa South Offshore	Koitiata
Status:	GRANTED	SUBMITTED	SUBMITTED	SUBMITTED
Granted:	19/12/2012			
Commenced:	19/12/2012			
Received:		12/03/2012	12/03/2012	12/03/2012
Duration:	5 years	5 years	5 years	5 years
Expires:	18/12/2017			
Area:	143070 HECTARE	87652.6 HECTARE	92766.5 HECTARE	133361.7 HECTARE
Minerals:	Aluminium	Aluminium	Aluminium	Aluminium
	Antimony	Antimony	Antimony	Antimony
	Bismuth	Bismuth	Bismuth	Bismuth
	Copper	Copper	Copper	Copper
	Garnet	Garnet	Garnet	Garnet
	Gold	Gold	Gold	Gold
	Ilmenite	Ilmenite	Ilmenite	Ilmenite
	Iron	Iron	Iron	Iron
	Ironsand	Ironsand	Ironsand	Ironsand
	Lead	Lead	Lead	Lead

	No. 54068	No. 54270	No. 54271	No. 54272
	Magnesium	Magnesium	Magnesium	Magnesium
	Molybdenum	Molybdenum	Molybdenum	Molybdenum
	Nickel	Nickel	Nickel	Nickel
	Platinum Group Metals	Platinum Group Metals	Platinum Group Metals	Platinum Group Metals
	Rare Earths	Rare Earths	Rare Earths	Rare Earths
	Rutile	Rutile	Rutile	Rutile
	Silver	Silver	Silver	Silver
	Tantalum	Tantalum	Tantalum	Tantalum
	Tin	Tin	Tin	Tin
	Titanium	Titanium	Titanium	Titanium
	Tungsten	Tungsten	Tungsten	Tungsten
	Zinc	Zinc	Zinc	Zinc
	Zircon	Zircon	Zircon	Zircon

Table 2-2: TTRL Continental Shelf Licence and Mining Permit

	No. 50753	No. 55581
Commodity:	MINERALS	MINERALS
Type:	Continental Shelf Licence	Mining Permit
Owners:	TTRL	TTRL
Location:	Taranaki	Taranaki
Operation Name:	Offshore Taranaki	South Taranaki Bight
Status:	GRANTED	SUBMITTED
Granted:	17/12/2010	
Commenced:	17/12/2010	
Received:		30/07/2013
Duration:	4 years	20 years
Expires:	16/12/2014	
Area:	3314 SQKM	6575.9 HECTARE
Minerals:	Aggregate	Aluminium
	Aluminium	Antimony
	Andesite	Apatite
	Antimony	Argilllite
	Apatite	Bentonite
	Argillite	Bismuth
	Basalt	Clay – high quality
	Bentonite	Clay – low quality
	Bismuth	Copper
	Clay – High Quality	Diamond
	Clay – Low Quality	Dolomite
	Coal – Hard/Semi-Hard Coking	Feldspar
	Coal – Lignite	Garnet
	Coal – Thermal/Semi-Soft Coking	Gemstones
	Conglomerate	Gold
	Copper	Ilmenite
	Dacite	Iron
	Decorative Pebbles	Ironsand
	Decorative Stone	Kauri Gum
	Diamond	Lead
	Diatomite	Magnesium
	Dolomite	Molybdenum
	Dunite	Monazite
	Feldspar	Nickel
	Fireclay	Phosphate

	No. 50753	No. 55581
	Garnet	Platinum Group Metals
	Gemstones	Quartz
	Gold	Rare Earths
	Granite	Rutile
	Gravel	Silica
	Ignimbrite	Silver
	Ilmenite	Sulfur
	Iron	Talc
	Ironsand	Tantalum
	Kauri gum	Tin
	Lead	Titanium
	Limestone	Topaz
	Magnesium	Tungsten
	Marble	Zinc
	Marl	Zircon
	Molybdenum	
	Monazite	
	Mudstone	
	Nickel	
	Peat	
	Perlite	
	Phosphate	
	Platinum Group Metals	
	Pumice	
	Quartz	
	Rare Earths	
	Rhyolite	
	Rutile	
	Sand	
	Sandstone	
	Schist	
	Scoria	
	Serpentinite	
	Silica	
	Silica Sand	
	Siltstone	
	Silver	
	Slate	
	Sulfur	

	No. 50753	No. 55581
	Talc	
	Tantalum	
	Tin	
	Titanium	
	Topaz	
	Tuff	
	Tungsten	
	Volcanic Ash	
	Zeolite	
	Zinc	
	Zircon	

Table 2-3: Permit Types (Verbatim – *NZPaM, 2013*)

	Prospecting Permit	Exploration Permit	Mining Permit
Purpose	To identify land likely to contain exploitable deposit	To identify deposits and evaluate the feasibility of mining	Economic recovery of an identified resource
Which Permit	No previous work has been done to locate a possible deposit	Often applied for first, rather than a prospecting permit, when higher impact work intended	The nature and extent of the mineable mineral resource or exploitable mineral deposit are known accurately
Activities	Very low impact, e.g. literature search, geological mapping, hand sampling or aerial surveys	May include literature review, drilling, bulk sampling and mine feasibility studies	Mineral extraction
Allocation	Acceptable work program offer	Acceptable work program offer	Acceptable work program offer
		As a subsequent exploration permit following a prospecting permit	As a subsequent mining permit following an exploration permit
		Newly available acreage (NAA) – competitive permit allocation process over available land following permit expiry, surrender, revocation or relinquishment	Newly available acreage (NAA) – competitive permit allocation process over available land following permit expiry, surrender, revocation or relinquishment
		Competitive Tender	
Exclusive Right	Yes, unless non-exclusive permit sought	Yes	Yes
Duration	2 years	5 years	Up to 40 years but commonly under 20 years (related to extent of reserves and resources and work program)
	Renewal up to another 2 years	Renewal up to another 5 years over half of area	
		Appraisal extension possible	
Size	No upper limit – relates to work program	No upper limit – relates to work program	Related to extent of discovery and work program

	Prospecting Permit	Exploration Permit	Mining Permit
Royalties	Not applicable	Not applicable unless annual production is greater than $200 000 in value	For permits under the 1996 minerals programs, 1% ad valorem royalty (AVR) for net sales revenues up to $1 million per annum. Where net sales revenues over $1 million pa, higher of either 1% AVR or 5% accounting profits royalty
			For permits under the 2008 minerals program: specific rate royalty (SRR) for low value to volume minerals and tiered AVR for precious metals and platinum group elements
Legislation	Crown Minerals Act 1991		
	Crown Minerals (Minerals and Coal) Regulations 2007		
	Crown Minerals (Minerals Fees) Regulations 2006		
	Minerals Program for Minerals (other than coal & petroleum) 1996		
	Minerals Program for Coal 1996		
	Minerals Program for Minerals (Excluding Petroleum) 2008		
Land Access Rights	For minimum impact activity, 10 days' notice except over special classes of land, e.g. urban conservation land		
	For other exploration and mining, land access arrangement with landowner and occupier		

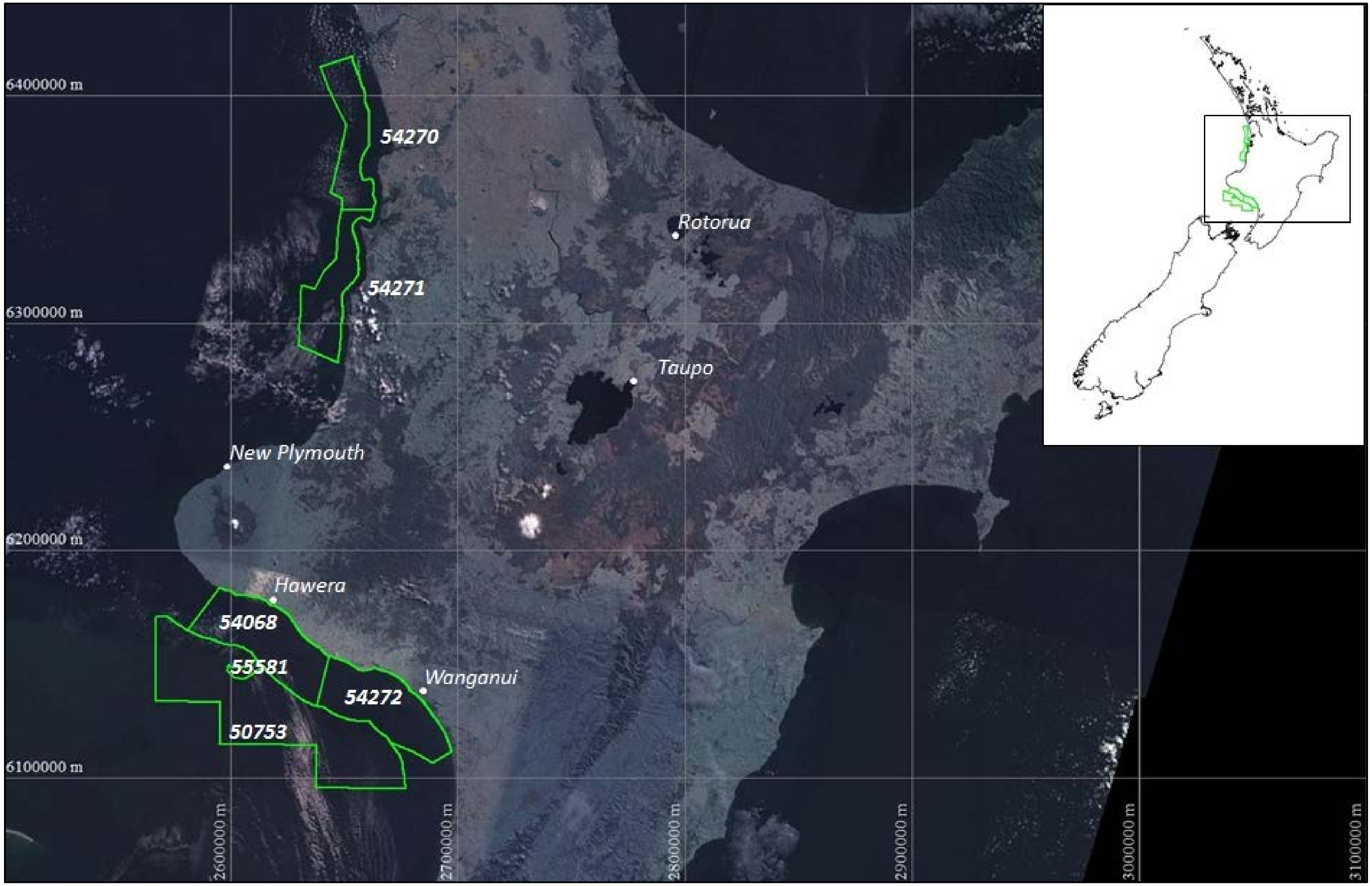


Figure 2-2: TTRL Permits and Licences (Crown Minerals, 2013; LANDSAT Imagery)

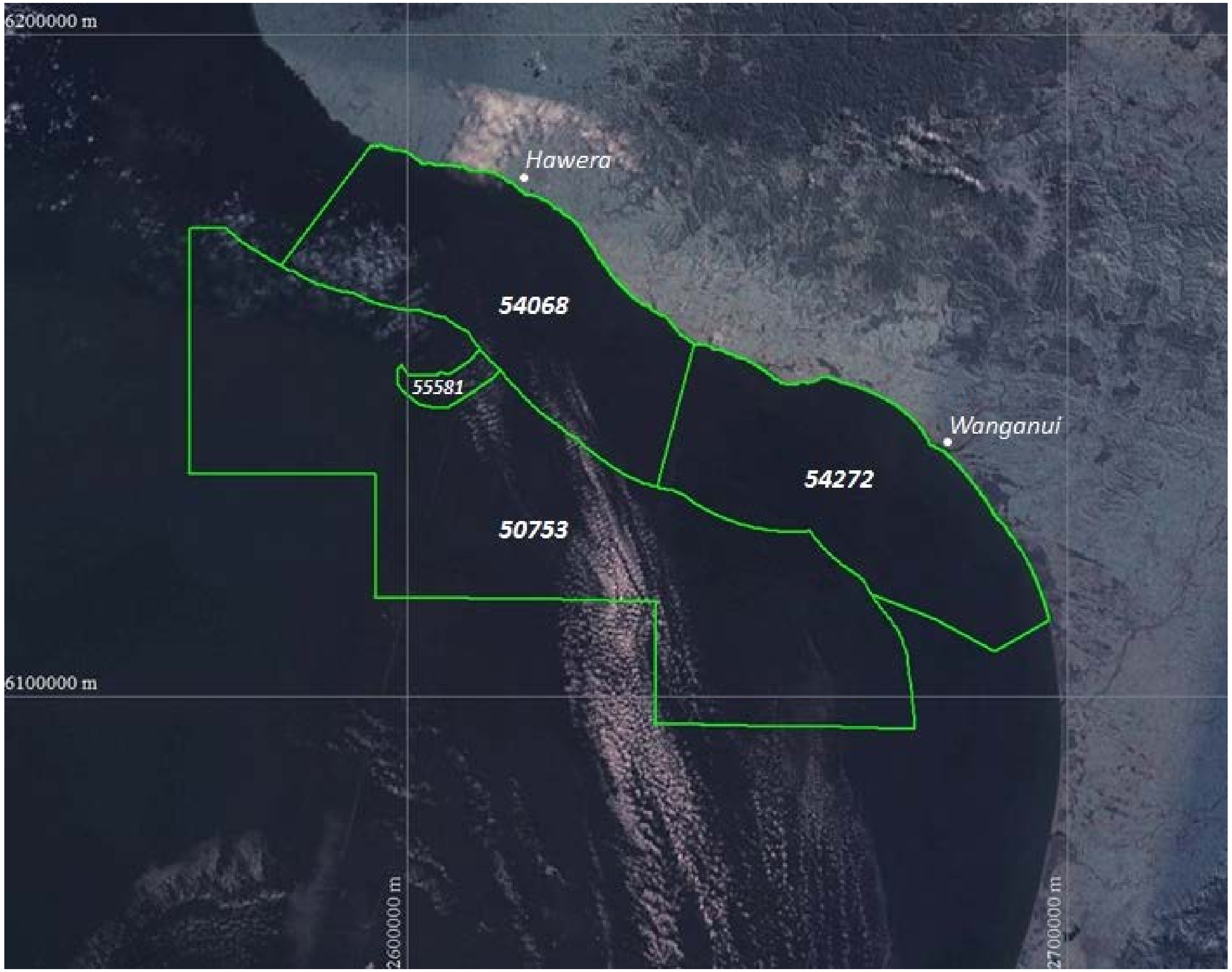


Figure 2-3: TTRL Mining Lease Application (Crown Minerals, 2013; LANDSAT Imagery)

2.1.1 Continental Shelf Licence

The Continental Shelf Act 1964 (CSA) makes provision for the exploration and exploitation of the continental shelf of New Zealand. It provides for the granting of licences in relation to prospecting and mining of minerals on the continental shelf, and the carrying out of operations for the recovery of minerals.

The jurisdiction of the CSA is the seabed and subsoils of those marine areas beyond the 12 nautical mile limit of New Zealand to a distance of 200 nautical miles and in some areas to the outer edge of the continental margin. (Crown Minerals, 2013, Crown Minerals, 2013A)

CSA50753 allows for detailed exploration and sampling during the first four years of tenure and detailed test pitting and bulk sampling during the second four years. The licence generally follows the Model Clauses for Prospecting Activity (Crown Minerals, 2013C) but includes detailed environmental base line studies and ongoing monitoring (Appendix A).

2.1.2 Royalties

Permit 55581 is a Tier 1 mining permit under the New Zealand 2013 Royalty regime. Under the Crown Minerals (Royalties for Minerals Other than Petroleum) Regulations 2013 (2013/206) the holder of a Tier 1 exploration or mining permit must pay the higher of—

a) an *ad valorem* royalty of 2% of the net sales revenue of the minerals obtained under the permit; and

b) an accounting profits royalty of 10% of the accounting profits, or provisional accounting profits, as the case may be, of the minerals obtained under the permit.

For permit holders subject to the 2013 royalty regime, no royalty is payable for Tier 1 permit holders (and Tier 2 coal permit holders) if the net sales revenues are less than $200 000 for the year (or average less than $16 666 per month if the reporting concerned is less than a year). For Tier 2 permit holders other than coal permit holders, no royalty is payable if the royalty calculated as payable is less than $2000 for the year (or averages less than $167 per month for the interim royalty reporting period) (NZPAM).

2.2 Previous Work

2.2.1 GNS

In 2008, TTRL commissioned the Institute of Geological and Nuclear Sciences Limited (GNS) to carry out desktop studies on the South Taranaki prospect. GNS produced two reports; Graham (2008) undertook an extensive literature search and produced a summary document and detailed bibliography. Christie and Jones (2008) compiled a comprehensive GIS data set to facilitate TTRL's ongoing exploration.

2.2.2 Fugro

In 2009 Fugro Airborne Surveys Pty Ltd processed the existing historical airborne magnetic survey data for the TTRL permits and created a 3D inversion model of the ironsand deposits. The airborne magnetic dataset, once filtered appropriately, was inverted to create a model of magnetic susceptibilities. These magnetic susceptibilities were transformed into volumes of percent magnetite based on the specific magnetic susceptibility of magnetite at cut-off grades of 1% to 6% and 12% (Cameron, 2009).

Table 2-4 details the results of that exercise. For example at a 5% magnetite cut-off the Fugro model contains 1.3 billion (1.3 × 10^9) cubic metres of sand with an average grade of 9.3% magnetite.

Table 2-4: 2009 Fugro – Predicted Tonnes and Grade from Geophysics

Cut-Off Grade %magnetite	Depth Slice	Volume m^3	Average Grade %magnetite
1%	0 to 40 m	10 161 200 000	3.0
1%	0 to 10 m	3 395 300 000	3.3
1%	10 m to 20 m	2 876 000 000	3.1
1%	20 m to 30 m	2 239 800 000	2.8
1%	30 m to 40 m	1 650 100 000	2.6
2%	0 to 40 m	4 881 100 000	4.8
2%	0 to 10 m	1 841 300 000	4.9
2%	10 m to 20 m	1 406 200 000	4.8
2%	20 m to 30 m	985 600 000	4.6
2%	30 m to 40 m	648 000 000	4.5
3%	0 to 40 m	2 890 600 000	6.4
3%	0 to 10 m	1 139 300 000	6.5
3%	10 m to 20 m	830 100 000	6.4
3%	20 m to 30 m	561 300 000	6.3
3%	30 m to 40 m	359 900 000	6.1
4%	0 to 40 m	1 909 600 000	7.9
4%	0 to 10 m	770 400 000	7.9
4%	10 m to 20 m	546 200 000	8.0
4%	20 m to 30 m	367 300 000	7.8
4%	30 m to 40 m	225 700 000	7.7
5%	0 to 40 m	1 352 900 000	9.3
5%	0 to 10 m	547 800 000	9.4
5%	10 m to 20 m	384 600 000	9.4
5%	20 m to 30 m	259 700 000	9.2
5%	30 m to 40 m	160 800 000	9.0
6%	0 to 40 m	997 400 000	10.7
6%	0 to 10 m	400 900 000	10.8
6%	10 m to 20 m	286 800 000	10.8
6%	20 m to 30 m	191 600 000	10.6
6%	30 m to 40 m	118 100 000	10.2
12%	0 to 40 m	277 000 000	17.2
12%	0 to 10 m	112 000 000	17.5
12%	10 m to 20 m	84 000 000	17.1
12%	20 m to 30 m	52 600 000	17.0
12%	30 m to 40 m	28 400 000	16.5

In 2010 Fugro flew a high resolution magnetic survey over the TTRL permits and rebuilt the 3D inversion model.

Table 2-5 and Table 2-6 summarise the Fugro resource estimate at a range of cut-off grades. Table 2-7 summarises the Fugro Resource within the limits of the Golder resource model for Area 2 (Figure 6-3, Figure 7-1). The Fugro figures are underestimated as they exclude part of the south of the Golder model. Table 2-8 is Fugro's summary of the Golder resource for the same area. The Golder figures are from the 2010 resource estimate and are significantly less than those from the 2011 resource estimates.

Table 2-5: Estimated Northern Permit Resource (Fugro)

Cut-Off Grade	Depth Slice	Volume	Titano-magnetite	Bulk Tonnage	Titano-magnetite	Contained Fe
(% titanomagnetite)		(Mm^3)	%	(Mt)	(Mt)	(Mt)
5	0 to 60 m	9.36	7.5	18.73	1.41	0.85
10	0 to 60 m	1.21	14.3	2.43	0.35	0.21
15	0 to 60 m	0.36	20.2	0.72	0.15	0.09
20	0 to 60 m	0.14	25.1	0.28	0.07	0.04

Table 2-6: Estimated Southern Permit Resource (Fugro)

Cut-Off Grade	Depth Slice	Volume	Titano-magnetite	Bulk Tonnage	Titano-magnetite	Contained Fe
(% titanomagnetite)		(Mm^3)	%	(Mt)	(Mt)	(Mt)
5	0 to 60 m	12.54	7.1	25.08	1.79	1.07
10	0 to 60 m	1.30	13.5	2.60	0.35	0.21
15	0 to 60 m	0.30	19.4	0.60	0.12	0.07
20	0 to 60 m	0.08	25.8	0.16	0.04	0.03

Table 2-7: Fugro Resource – Golder Area 2 Model Limits

Cut-Off Grade	Depth Slice	Contained Fe
(% titanomagnetite)		(kt)
5	0 to 6 m	55.427
10	0 to 6 m	15.535
15	0 to 6 m	5.291
20	0 to 6 m	1.945

Table 2-8: Golder Area 2 Resource

Cut-Off Grade	Depth Slice	Contained Fe
(% titanomagnetite)		(kt)
5	0 to 6 m	60.294
10	0 to 6 m	20.609
15	0 to 6 m	3.442
20	0 to 6 m	0.885

2.2.3 Golder

The preliminary objective of the original TTRL drilling campaign was to validate the Fugro resource. A correlation between the Fugro results and the TTRL drilling could not be achieved. As discussed later, the geological interpretation has a layer of reworked sands overlying a series of palaeo-geomorphological features. The current interpretation concludes that most of the drilling is only sampling the overlying sands.

In 2010 Golder reported an inferred mineral resource estimate for the South Taranaki ironsand deposit in accordance with the guidelines of the JORC Code (JORC, 2004). Table 2-9 details the 2010 estimate averaging 1.04 billion tonnes (1.04 × 10^9) with an average grade of 5.88% Fe. The resource was reported at a 5% Fe2O3 head grade cut off. The economic assessment at the time indicated this cut off would provide an viable head grade for the processing plant.

Table 2-9: 2010 Golder *In Situ* Mineral Resource at 5% Fe_2O_3 Cut-Off (All Grades are %)

Class	Zone	Mt	Fe_2O_3	Fe*	Al_2O_3	SiO_2	TiO_2	P_2O_5	LOI	Rec**
Inferred	1	143	10.59	7.41	9.36	49.14	1.07	0.16	3.04	91.71
	2	146	6.75	4.72	10.68	40.44	0.68	0.17	4.48	83.52
	4	224	7.40	5.18	9.61	38.42	0.71	0.19	5.61	84.52
	5	22	10.82	7.57	10.36	39.47	1.07	0.21	1.21	83.00
	6	145	9.50	6.64	11.85	50.50	1.00	0.17	2.73	93.35
	7	205	8.51	5.95	9.32	40.23	0.82	0.17	5.51	87.95
	8	155	7.88	5.51	8.13	50.26	0.69	0.11	4.17	91.37
Grand Total		**1040**	**8.40**	**5.88**	**9.77**	**44.01**	**0.82**	**0.17**	**4.37**	**88.27**

*Fe calculated from Fe_2O_3 × 0.6994; **physical mass recovery after 2 mm screening.

In July 2011 the South Taranaki ironsand resource was updated to include additional extensional and infill drilling. An Indicated plus Inferred Mineral Resource of 2.12 billion tonnes (2.12 × 10^9) at 5.64% Fe was reported (Table 2-10).

Table 2-10: July 2011 Golder *In Situ* Mineral Resource at 5% Fe_2O_3 Cut-Off (All Grades are %)

Class	Zone	Mt	Fe_2O_3	Fe*	Al_2O_3	SiO_2	TiO_2	P_2O_5	LOI	Rec.**
Indicated	1	97	9.83	6.87	9.80	51.71	1.00	0.16	2.85	93.01
	2	65	7.69	5.38	12.69	44.02	0.78	0.21	1.80	86.14
	6	124	10.18	7.12	10.87	50.54	1.07	0.18	2.81	92.90
	7	438	7.46	5.22	13.16	47.79	0.77	0.21	2.48	90.76
Indicated Total		***724***	***8.27***	***5.78***	***12.27***	***48.45***	***0.85***	***0.20***	***2.52***	***91.01***
Inferred	1	86	14.97	10.47	7.84	42.80	1.49	0.22	3.68	91.28
	2	481	6.93	4.85	12.38	43.68	0.72	0.20	3.75	86.96
	4	189	7.49	5.24	9.57	38.39	0.72	0.19	5.55	84.55
	5	77	7.24	5.06	12.71	43.47	0.73	0.23	1.65	84.43
	6	125	9.29	6.50	11.67	51.43	0.97	0.18	3.69	96.16
	7	285	7.54	5.27	9.96	40.29	0.74	0.19	5.13	85.88
	8**	155	7.88	5.51	8.13	50.26	0.69	0.11	4.17	91.37
Inferred Total		***1397***	***7.96***	***5.56***	***10.71***	***43.63***	***0.79***	***0.19***	***4.20***	***87.85***
Grand Total		**2121**	**8.06**	**5.64**	**11.24**	**45.27**	**0.81**	**0.19**	**3.63**	**88.93**

*Fe calculated from Fe_2O_3 × 0.6994

In December 2011 the resource was updated for drilling undertaken up to 13 November 2011. An Indicated plus Inferred Mineral Resource of 4.6 billion tonnes (4.6 × 10^9) at 6.23% Fe was reported (Table 2-11). Note that the table reports recoverable tonnes excluding the +2 mm material screened off prior to analysis.

Table 2-11: Nov 2011 Recoverable Mineral Resource at 5% Fe2O3 Cut-Off Grade (All Grades are %)

Class	Zone	Mt Rec.	Fe_2O_3	Fe*	Al_2O_3	SiO_2	TiO_2	P_2O_5	LOI	Rec
Indicated	1	130	10.15	7.1	10.54	55.36	1.03	0.17	3.35	92.20
	2	182	7.63	5.34	14.36	50.04	0.79	0.24	4.58	86.41
	3	305	10.88	7.61	13.06	52.13	1.1	0.25	2.13	97.87
	5	107	7.44	5.2	14.78	52.19	0.78	0.24	3.97	90.63
	6	207	9.11	6.37	12.62	56.59	0.96	0.19	2.87	94.68
	7	500	8.11	5.67	14.65	52.97	0.83	0.24	2.63	90.78
	9	74	8.69	6.08	14.43	51.43	0.87	0.25	2.43	90.24
Indicated Total		***1504***	***8.91***	***6.23***	***13.66***	***53.02***	***0.91***	***0.23***	***2.95***	**92.31**
Inferred	1	119	16.93	11.84	8.41	46.88	1.69	0.25	3.78	93.47
	2	431	7.74	5.41	14.28	50.14	0.8	0.23	4.68	87.69
	3	469	11.22	7.85	13.58	51.06	1.14	0.26	2.51	96.69
	4	213	8.56	5.99	10.86	44.27	0.82	0.22	7.82	83.66
	5	8	7.17	5.01	13.94	53.37	0.75	0.24	3.45	83.38
	6	577	9.43	6.59	13.09	53.86	0.98	0.21	3.02	94.64
	7	374	8.41	5.88	11.62	46.9	0.83	0.22	6.5	85.39
	8**	374	7.13	4.99	9.59	56.49	0.65	0.14	5.46	90.96
	9	568	7.28	5.09	16.25	53.15	0.75	0.25	2.16	91.79
Inferred Total		***3134***	***8.9***	***6.22***	***12.98***	***51.37***	***0.9***	***0.22***	***4.08***	**91.13**
Grand Total		**4638**	**8.9**	**6.23**	**13.2**	**51.9**	**0.9**	**0.22**	**3.71**	**91.51**

*Fe calculated from Fe_2O_3 × 0.6994

In December 2012 the resource was updated to include drilling undertaken up to 20 November 2012. An Indicated plus Inferred Mineral Resource of 4.6 billion tonnes (4.6 × 10^9) at 6.25% Fe (8.94 Fe_2O_3) was reported (Table 2-12). Note that the table reports recoverable tonnes.

Table 2-12: Nov 2012 Recoverable Mineral Resource at 5% Fe2O3 Cut-Off Grade (All Grades are %)

Class	Zone	Mt	Fe_2O_3	Fe	Al_2O_3	SiO_2	TiO_2	P_2O_5	LOI	Rec
Indicated	1	131.3	9.65	6.75	10.71	56.14	0.98	0.171	0.17	92.21
	2	184.1	7.69	5.38	14.46	50.31	0.8	0.241	0.17	86.85
	3	426.8	11.98	8.38	12.72	51.18	1.21	0.259	0.21	98.06
	5	108.8	7.32	5.12	15.05	52.37	0.78	0.24	0.15	90.8
	6	220.7	9.86	6.90	12.54	55.52	1.03	0.197	0.17	94.96
	7	501.3	8.05	5.63	14.65	52.99	0.83	0.234	0.17	90.67
	9	150.5	8.65	6.05	14.2	52.21	0.87	0.248	0.18	94.69
Total Indicated		***1723.4***	***9.35***	6.54	***13.57***	***52.71***	***0.96***	***0.233***	***0.18***	***93.12***
Inferred	1	157.8	14.76	10.32	10.06	48.79	1.5	0.247	0.22	92.81
	2	429.4	7.65	5.35	14.31	49.86	0.8	0.227	0.16	86.86
	3	355.3	9.33	6.53	14.43	52.7	0.96	0.25	0.18	96.13
	4	212.1	8.55	5.98	11.01	44.61	0.82	0.225	0.19	83.4
	5	7.4	6.68	4.67	13.46	53.5	0.7	0.224	0.14	89.26
	6	617.4	9.63	6.74	13.16	53.86	1	0.215	0.17	94.74
	7	375.3	8.41	5.88	11.72	47.2	0.83	0.219	0.18	85.58
	8	191.6	7.04	4.92	9.68	56.77	0.64	0.141	0.14	90.83
	9	590.5	7.33	5.13	16.29	53.13	0.76	0.253	0.16	91.75
Total Inferred		***2936.8***	***8.71***	6.09	***13.38***	***51.38***	***0.89***	***0.227***	***0.17***	***90.79***
Grand Total		**4660.2**	**8.94**	6.25	**13.45**	**51.88**	**0.91**	**0.229**	**0.17**	**91.65**

3.0 DATA VALIDATION

3.1 Site Visits

Golder's representative visited the site between 28 and 31 January, 2010. Stephen Godfrey, Associate, and Principal Resource Geologist with Golder's Perth office travelled to New Zealand and was escorted by TTRL representative, Paul Vermeulen, for most of the visit. The site visit included a detailed review of the drilling methodology, sample collection and handling.

From 24 to 27 July 2012 Stephen Godfrey and James Farrell (Associate, Senior Geologist) visited the TTRL Wellington office and Porirua warehouse. The purpose of the visit was to review the project status, audit the analytical laboratory and review the pilot plant operation.

In February 2013 TTRL resumed drilling in the South Taranaki Bight offshore from Patea in New Zealand. The drilling was being undertaken using their deep drilling rig commissioned in 2012. Due to inclement weather Golder was unable to observe any of the six holes drilled in 2012. In order to review the deep drilling process a site visit was arranged by TTRL for Stephen Godfrey between 18 and 22 February 2013.

3.2 Observations

TTRL has undertaken a program of offshore sampling using the services of New Zealand Diving and Salvage (NZDS). The sampling program has included sediment sampling onshore and offshore. Preliminary investigation commonly involved lowering a magnet to the sea floor to identify the presence of magnetic minerals. Within the permit areas the return of magnetic sands from this process was almost ubiquitous. These grab samples are non-representative of the deposit and have not been used in any analyses or estimations.

In partnership with NZDS, TTRL developed a drill sampling system capable of sampling the first six metres of the sea bed. The drill rig was diver-operated on the sea floor. The drilling employs a passive triple tube reverse circulation system. In December 2010 the system was upgraded enabling it to be hydraulically controlled from the surface with diver support if necessary. In September 2011 the system was upgraded enabling a maximum drill depth of nine metres (Figure 3-1, Figure 3-2 and Figure 3-3).

The drilling rig is transported to the drill site by a service vessel and lowered to the sea floor. The original system was diver operated and restricted to operating in less than 25 m of water. Below this depth decompression is required for the diver to return to the surface. The service vessels for this campaign did not carry decompression chambers. The upgraded system can operate in deeper water, with the deepest hole to date drilled at 65 m below surface. A maximum water depth of 80 m is possible with the current drill rig configuration.

The original diver-supported six metre system was used to drill the first 148 holes. A further 631 holes have been drilled with the diver-less system.

In order to ensure the effectiveness of the drill system and the veracity of the samples, in 2010 a Golder representative spent a day on the service vessel the *Shoman* observing the drilling of three holes in the Graham Banks area.

The drill system uses a 75.75 mm OD bit and 75 mm OD pipe (approximately NQ). The drill used a single rod with a 6 m stroke. On the sea floor the diver releases the drill rod which penetrates under its own weight with most of the work being done by the hydraulic cutting action of the bit. Water is pumped down the outer tube and air down the inner tube with angled jets creating both a cutting and venturi-type effect to raise the sample. Drilling through sands is quite smooth and effective. If the drill encounters shell beds penetration may stop. Originally, a blast of air was used to get through shell beds, however this resulted in abnormally large samples because the blast created a cavity which then collapsed.

Golder advised that these air blast samples should be flagged in the database and not used for any resource analysis work. The system later employed a hand-operated winch and now uses a hydraulic system to exert down force on the drill rod to assist in penetrating shell beds.

The returned samples are collected from the base of a cyclonic separator. The size of the samples is normally consistent with the size of hole being drilled. When the downward progress of the drill is stopped the system returns clean water to the cyclone indicating there is no contamination from material inflow and that the drill is returning only material from the drill hole.

The drill system has some issues with larger particles not returning in the system as there is no cutting bit to break them up. These larger particles make up a very small proportion of the material being sampled and should have no material impact on the resource. The envisaged dredging/processing system to mine a deposit such as this would screen out anything larger than 2 mm, so any contained mineralisation has no material impact on the resource.

In 2012 a new rig was developed and deployed with the ability to drill to a depth of 40 m (Figure 3-4, Figure 3-5). This rig is diver operated on the seafloor. The rig uses a similar system to a land based RC drilling rig carrying six removable drill rods in a carousel. To date 20 drill holes have been drilled to a maximum depth of 30 m with this system. Seventeen of these drill holes were drilled for resource definition purposes, however due to issues with potentially non–representative sampling only six of the drill holes have been used in the resource estimation.

The drill rig and divers were connected to the service vessel by umbilicals. The drill rig compressor and pump were on the service vessel and all samples returned by bull hose to a cyclone on the deck. The system included full video contact between the sea floor rig and the boat. Divers also had video and audio contact with the surface crew. Drilling is monitored by a drill supervisor on the boat.

In 2013 a site visit was undertaken to observe the deep drilling rig in operation. Golder spent two days on the service vessel the *PMG Pride* and observed the drilling of two holes, STH019RC and STH016RC. The drill system uses a rotary reverse circulation process behind a tri-cone roller bit similar to a standard land based system. The drill rig is diver operated on the sea floor and consequently has similar depth and dive time restrictions to the shallow drilling process (Figure 3-4 and Figure 3-5).

The drill system was observed to be effective however uncertainty exists over how representative the sample collected was. Consequently, the deep holes drilled in 2013 were deemed unsuitable for use in resource estimations until the veracity of the samples collected can be confirmed (Golder, 2013).

Drilling is weather dependant. The tenements are exposed to the storms of the ‘roaring forties’ that come across the Tasman Sea. During the worst storms even Wanganui harbour is unsafe for vessels.

A total of 799 drill holes have been collared. Of these 633 were used in the resource estimation. The excluded holes were removed for the following reasons; they were bulk sample drill holes, they were aborted and redrilled; sample problems were identified; or they were outside the resource model areas.

The Spectrachem laboratory was visited in 2010 and 2012. The sample processing and analysis system was inspected during both visits, with the 2012 visit focussed on the DTR samples. In both instances the laboratory was observed by Golder to be performing as expected.

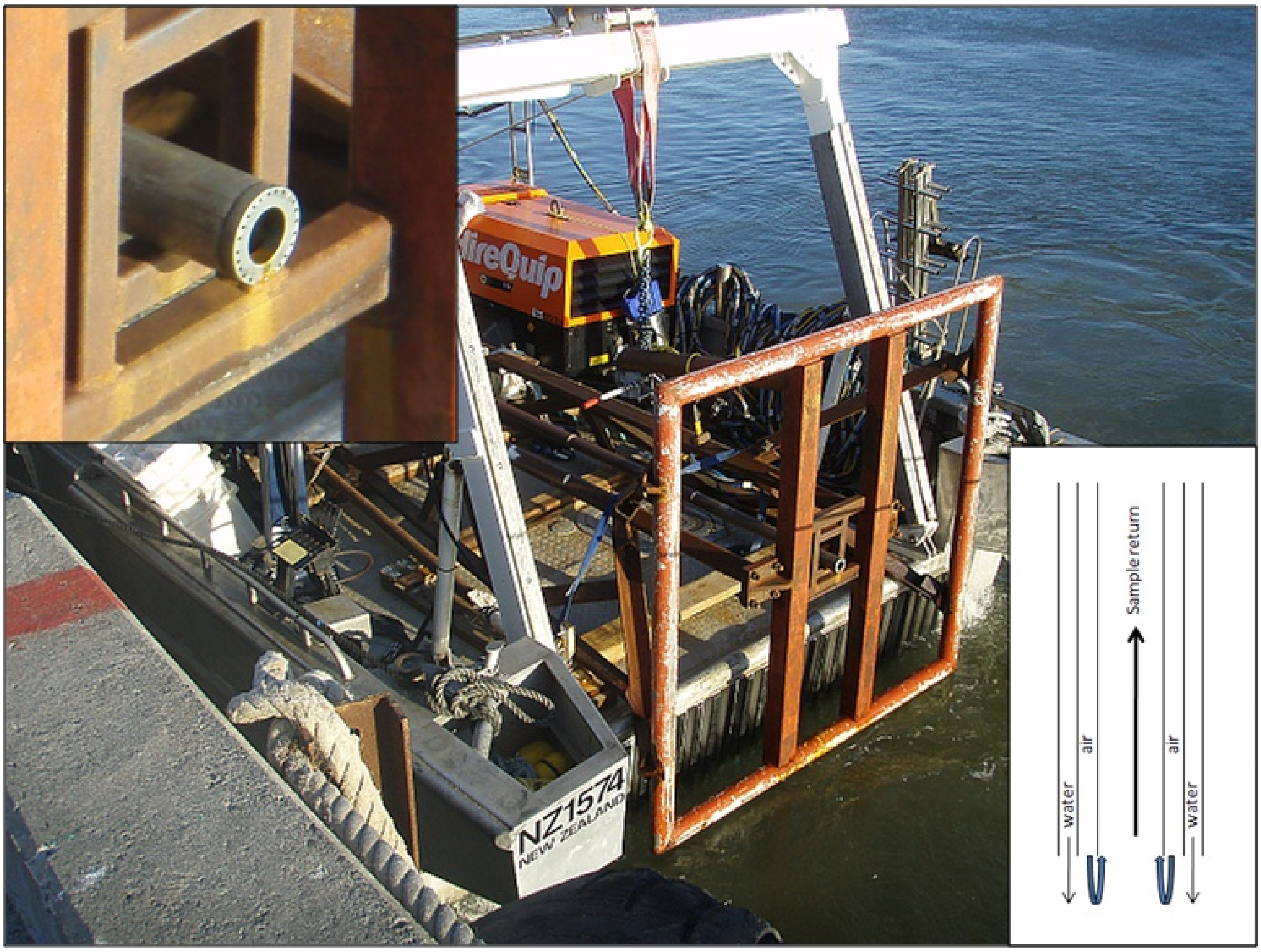


Figure 3-1: Drill Rig on the Shoman. Inset – Bit Detail and Circulation Diagram

Figure 3-2: Drilling is Diver Operated and Monitored from the Boat

Figure 3-3: Cyclone and Sample Collection

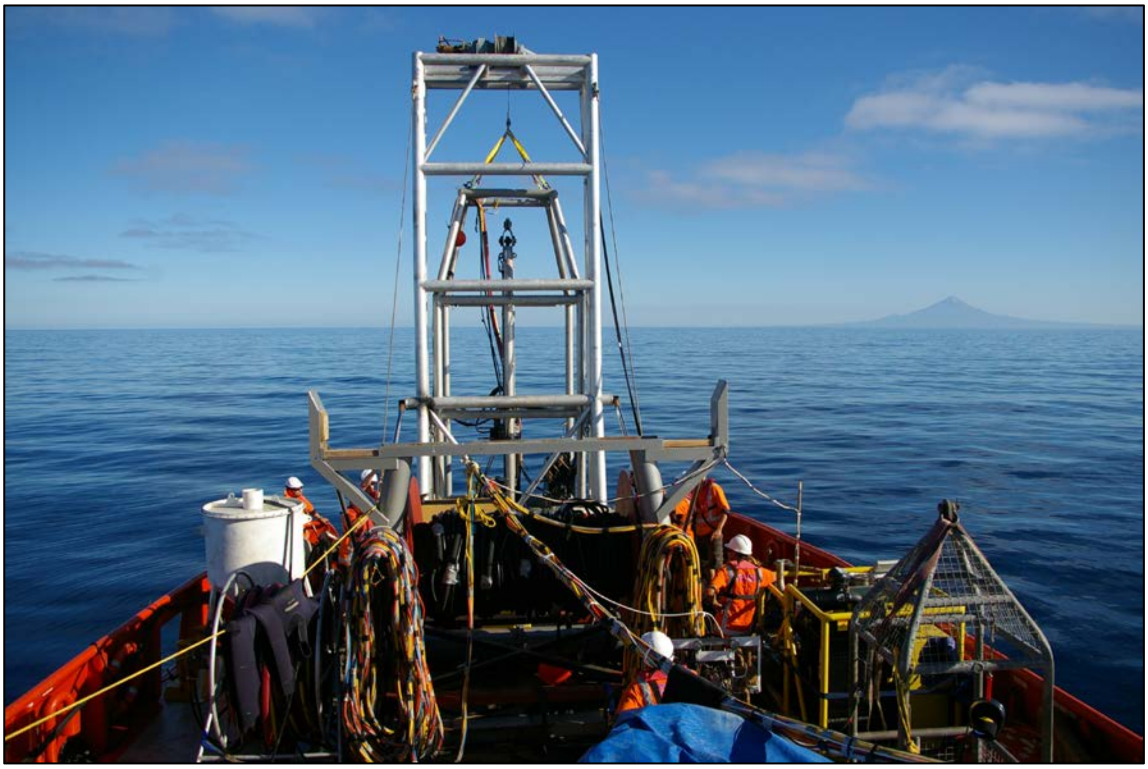


Figure 3-4: 30 m Drill Rig Being Deployed

Figure 3-5: PMG Pride – Drill Rig Deployed

3.3 Sampling

Samples are bagged, labelled clearly and stored on deck until the boat return to harbour. A preliminary log of the samples is made while at sea and a magnetic susceptibility reading is taken.

All samples are temporarily stored in Wanganui Port before being transported to the TTRL Porirua Warehouse. At the warehouse the samples are dried and split into eight. One split is sent for chemical analysis and another for geological logging. A field magnetic susceptibility reading is taken from the chemical analysis sample. The remaining splits are re-bagged and stored.

The sample for chemical analysis (head sample) is sent to Spectrachem for XRF analysis and returns the analyte suite listed in Table 3-1. The 2010 to 2011 samples for logging was sent to the National Institute of Water and Atmospheric Research (NIWA). Samples are now logged by TTRL geologists.

Note that the laboratory screens the sample to remove all material greater than 2 mm in diameter and records the recovery (%). This material, predominantly of shells and pebbles, is regarded as barren. The laboratory analysis is performed on the sub-2 mm material. The final model results need to take this into account. The model estimates the full volume and tonnes of the deposit so the estimated grades need to be diluted by the recovery.

In 2012 and 2013 selected samples were sent for Davis Tube Recovery (DTR) Analysis. The selected samples were from existing, and any new, drill holes in the proposed mining area. DTR analysis determines the magnetically recoverable portion of the sample by passing the sample through a high intensity magnetic field (Figure 3-6). The recovery is sensitive to the equipment set-up including particle size and magnetic intensity. The overall set-up is designed to emulate the eventual processing plant recovery on a laboratory scale. Some scale up factor may eventually be required in estimating an ore reserve. The recovered magnetic concentrate undergoes XRF analysis and returns the analyte suite as listed in Table 3-1. Note that the concentrate iron analysis returns Fe and the head analysis Fe_2O_3.

Table 3-1: Analysis Suites

Head	Concentrate
Al_2O_3	Al_2O_3
CaO	CaO
	As
	Co
	Cr
	Cu
Fe_2O_3	Fe
K_2O	K_2O
LOI	LOI
MgO	MgO
MnO	Mn
Na_2O	Na
	Ni
P_2O_5	P
	S
SiO_2	SiO_2
TiO_2	Ti
	V
	Zn

The holes drilled during 2010 were also analysed for a series of trace elements (Table 3-2). This has been discontinued in recent drilling. A small number of samples have also been analysed for Th, SO_3 and Cl. These analytes have not been included in any resource estimation.

Table 3-2: 2010 Trace Element Assay Suite

Trace Elements	
As	Rb
Ba	Sc
Ce	Sr
Cr	Th
Cu	U
Ga	V
La	Y
Nb	Zn
Ni	Zr
Pb	

Appendix B includes procedural documentation for TTRL's drill sampling program.

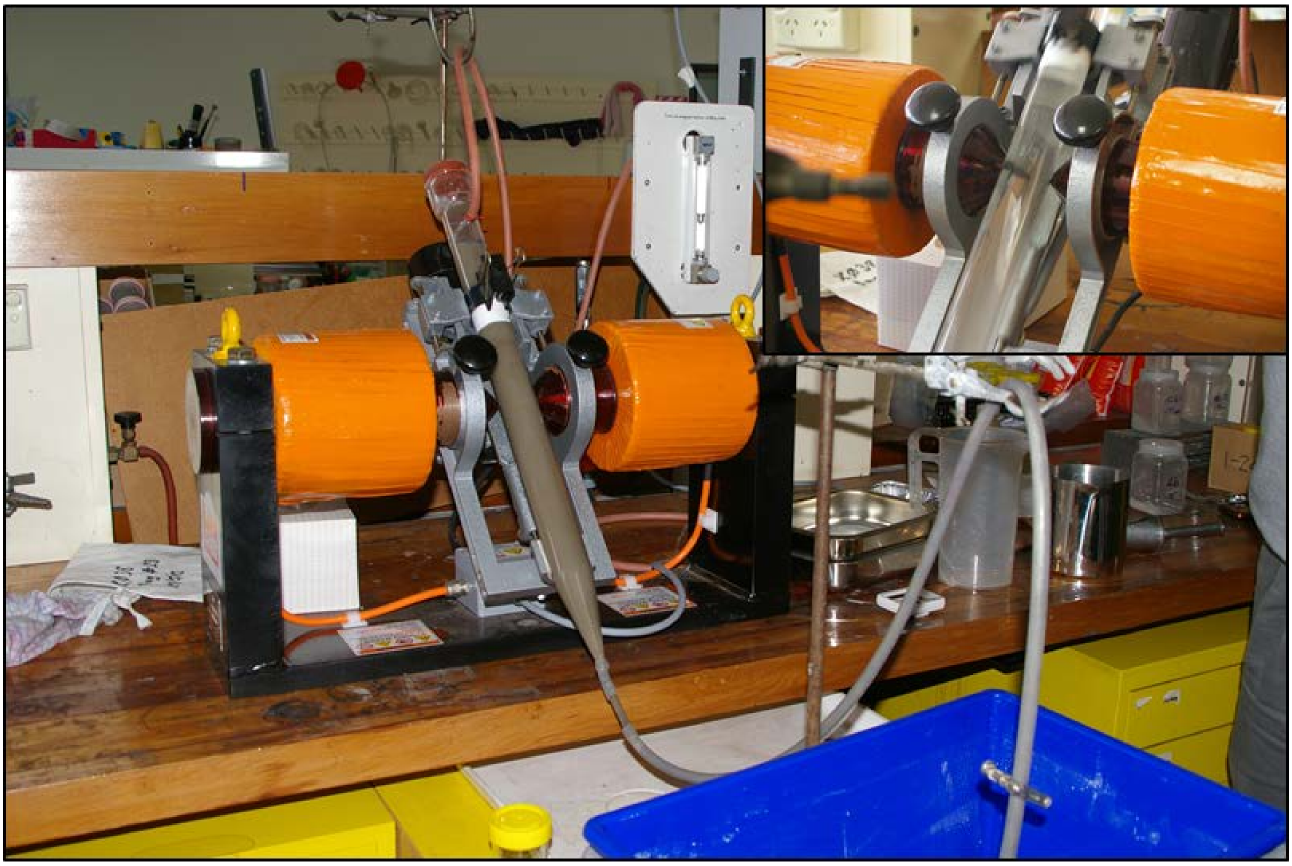


Figure 3-6: Main: Davis Tube. Inset: Recovered Concentrate

3.4 Pilot Plant

As part of the resource validation process the metallurgical pilot plant was observed operating during Golder's 2012 site visit. The pilot plant, a scaled down version of the anticipated final processing plant, is located at the Porirua warehouse. A bulk sample was collected from the proposed mining area for the pilot plant test work. The sample was obtained using the exploration drill rig. The pilot plant screens the sample at +20 mm then +2 mm with the sub-2 mm fraction going through a first pass Medium Intensity Magnetic Separation (MIMS) and Low Intensity Magnetic Separation (LIMS). The recovered concentrate is ground by ball mill to 53 µm (P80) and run through LIMS three times producing a final concentrate.
Figure 3-7 illustrates the pilot plant flowsheet at the time of inspection. Note that this design has been superseded.

JORC (2012) in defining a Mineral Resource requires that "there are reasonable prospects for eventual economic extraction". The successful production of concentrate by the pilot plant demonstrates that it is possible to the recover titanomagnetite from the South Taranaki ironsand deposits.

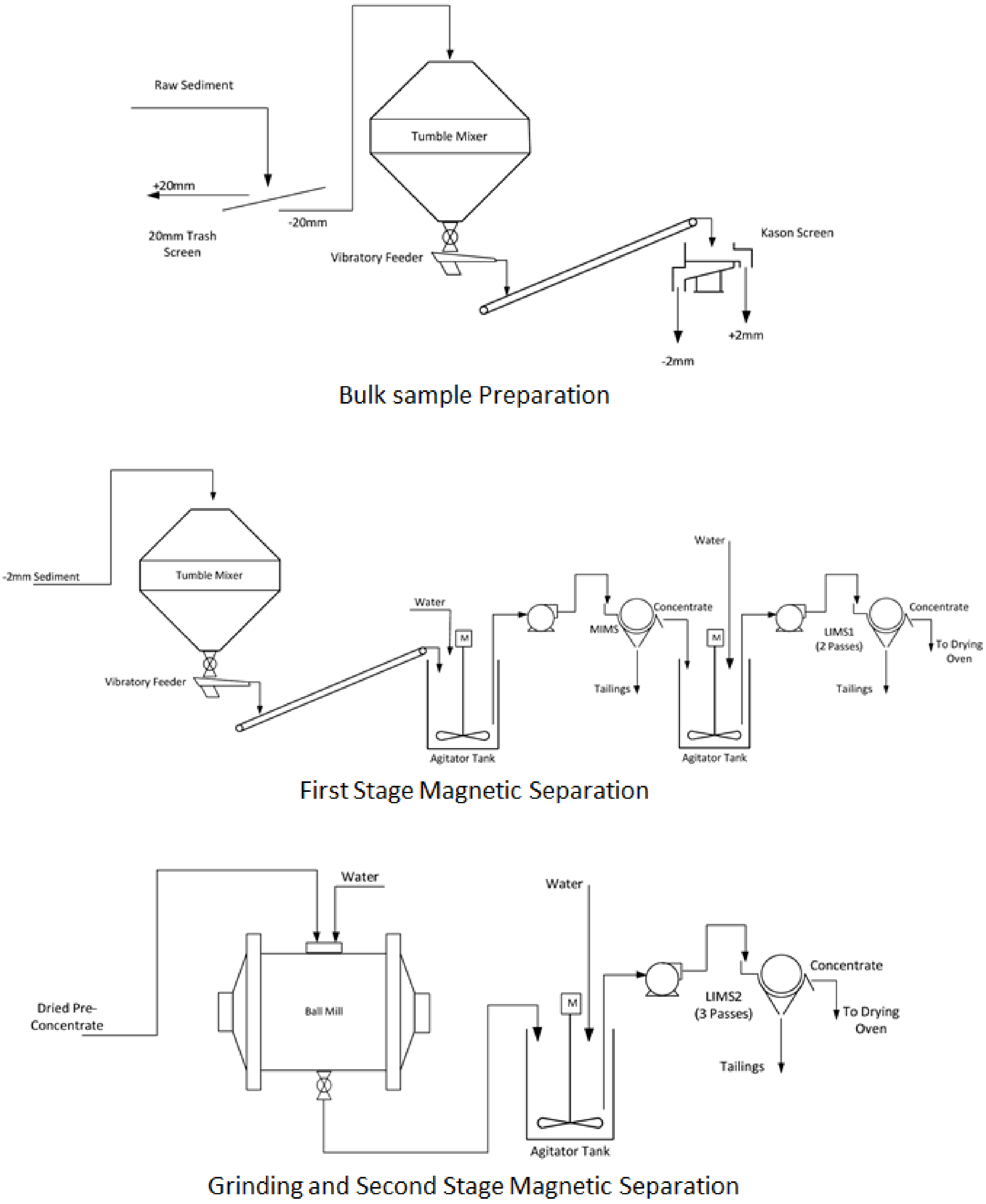


Figure 3-7: Pilot Plant Flowsheet (Superseded Design)

4.0 DATA PROVIDED TO GOLDER

In 2010 to 2011 Golder was provided with a comprehensive GIS data set. Topographic and bathymetric data was extracted from the GIS data set along with miscellaneous geographical information, e.g. coastlines, rivers and place names. The GIS data set also included magnetic geophysical imagery. The data provided for this resource update was the current geological drill hole database, and new bathymetric data acquired in the last 12 months. TTRL also provided documentation for their drilling, sampling and database procedures (Appendix B).

4.1 Review of Data Available for Resource Modelling

4.1.1 Data Locations

Drill hole locations are identified as the GPS location of the rear of the service vessel after the drill rig has been lowered to the sea floor. Readings are taken from a hand held GPS. During the site visits it was noted that the GPS unit was given little time to settle. This can improve the accuracy of the GPS reading but given the scale of the project this will have no material impact on any resource estimation.

The sea floor depth is normally the sonar depth. For the original drill program the diver also reported the reading from his depth gauge. If the sonar value was not available or not recorded the diver depth was used. No adjustment has been made for tides. This could have an impact of up to 3 m in this area. To eliminate any discrepancies caused by this all drill hole collars were registered to the bathymetric surface.

4.2 Database

The drilling data was supplied to Golder by TTRL for the November 2013 resource update as two MS Access databases, *TTR_DB1_To_Golders_20130903.accdb* and *DTR_Database.accdb.* The database contains drill results to 4 September 2013. The drill data was ported to the database *Golder_2013.accdb* where it was verified, filtered and exported to CSV files. The CSV files were loaded to the Vulcan Isis database *res2013_1709.ttr2.isix* for analysis and modelling.

Golder undertook a detailed audit of the TTRL database in July 2011. Only a few errors and discrepancies were noted and were corrected by TTRL.

The 2013 updated database was verified by Golder for internal integrity. The verification included:

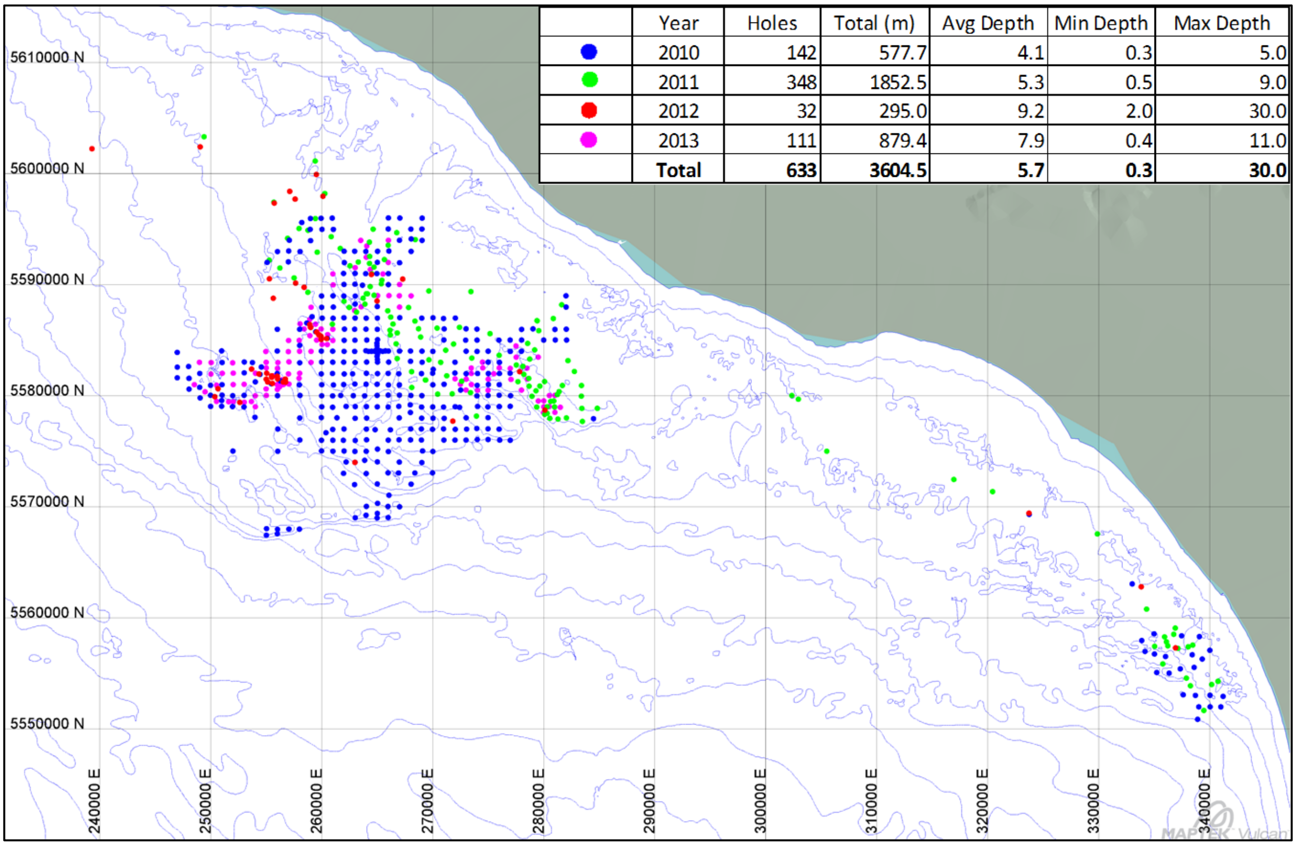
- Cross table checks (e.g. drill hole has collar information but no assay information).
- Final assay and geology depths validated against collar end of hole depth
- Check for overlapping intervals or gaps in the assay and geology tables.
- Check for duplicate drill hole names and/or duplicate coordinates.
- Check for null collar coordinates or coordinate values of zero.
- Check for integer coordinate values indicating of lack of detailed survey data.
- Extreme variations (≥10°) in drill hole azimuth or dip between consecutive downhole survey records.

No significant errors or discrepancies were identified by Golder, however 103 records were removed. These were grab samples, bulk samples and 42 drill holes. The drill holes were excluded because they

- were test holes,
- were not logged,
- were not sampled,
- had not had sample analysis returned,
- were twin holes (the other usually deeper twin was retained), or
- had incorrect survey data.

4.2.1 Drilling

The November 2013 resource update is based on 633 drill holes containing valid analytical results for 3653 samples representing 3604.5 m of drilled material. Figure 4-1 illustrates the locations of all drilling used in the resource estimate highlighted by drilling season.



	Year	Holes	Total (m)	Avg Depth	Min Depth	Max Depth
● (blue)	2010	142	577.7	4.1	0.3	5.0
● (green)	2011	348	1852.5	5.3	0.5	9.0
● (red)	2012	32	295.0	9.2	2.0	30.0
● (magenta)	2013	111	879.4	7.9	0.4	11.0
	Total	**633**	**3604.5**	**5.7**	**0.3**	**30.0**

Figure 4-1: Resource Drill Hole Locations – South Taranaki Ironsand Deposit

The majority (~42%) of holes are 5 m deep, the limit of the early drilling system. Approximately 26% of holes are less than 5 m and 32% of holes have been drilled with the newer rigs and are deeper than 5 m (Figure 4-2).

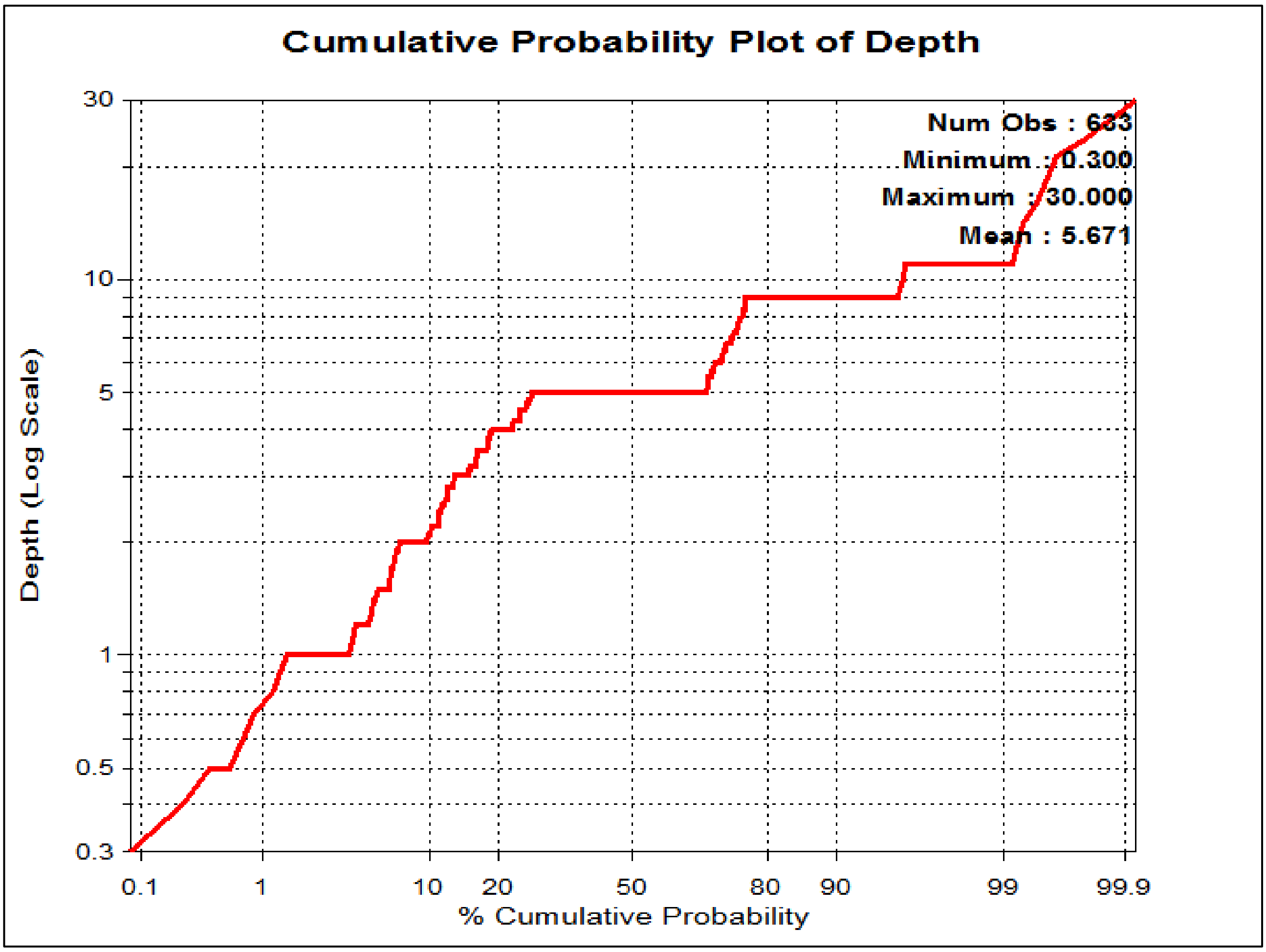


Figure 4-2: Resource Drill Hole Depths – South Taranaki Ironsand Deposit

Figure 4-3 illustrates the Area 2 model limits with the proposed mining area outlined. Within the proposed mining area 167 drill holes have had samples re-analysed for Davis Tube Recovery and the recovered concentrate analysed by XRF. Table 4-1 summarises the number of drill holes and samples available for each resource model area for resource estimation.

Table 4-1: Model Area Data

	Head Analysis		DTR/DTC Analysis	
	Drill Holes	Samples	Drill Holes	Samples
Area 2	589	3441		
Koitiata	44	212		
Proposed Mine Area			167	1289

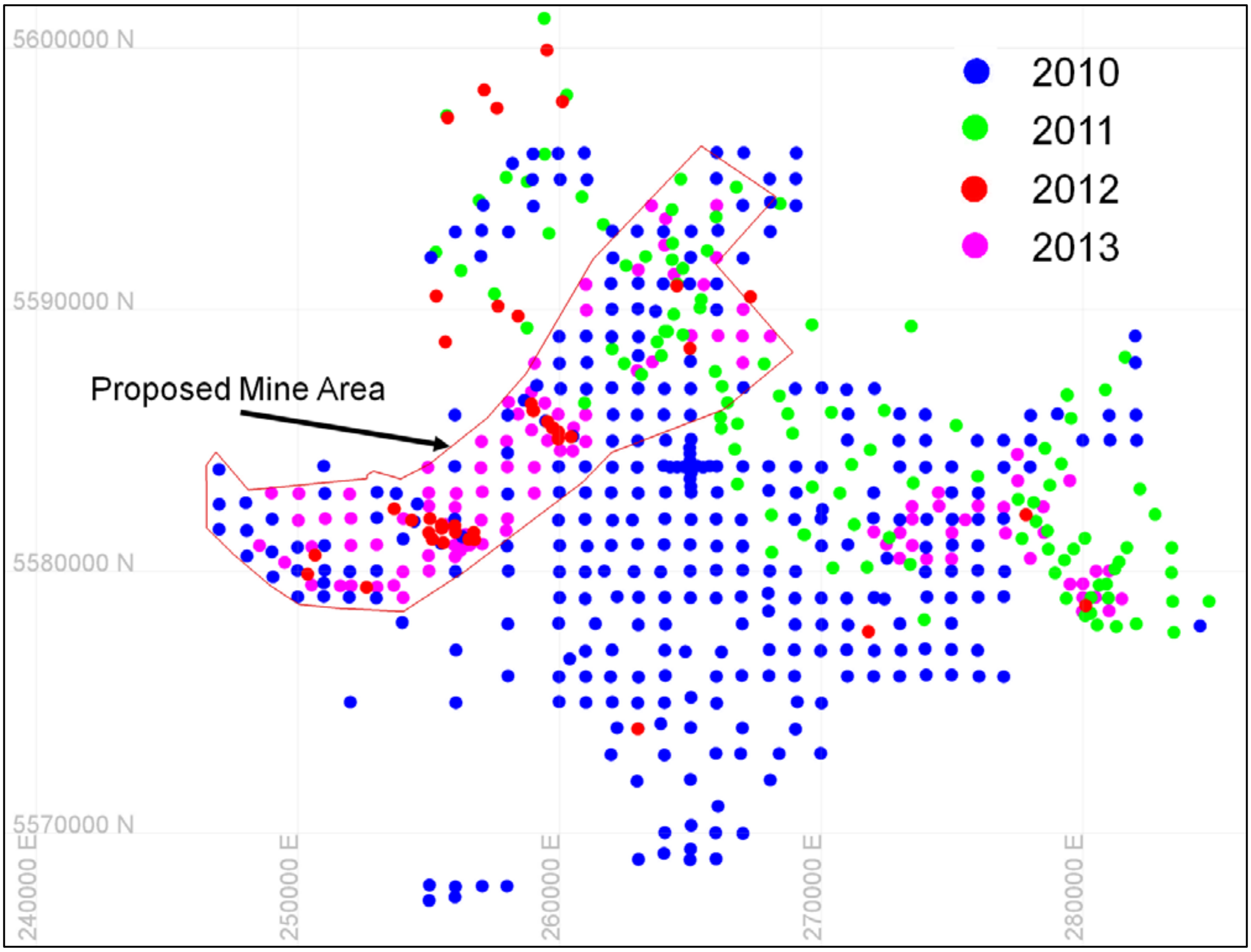


Figure 4-3: Drill Holes in the Proposed Mining Area

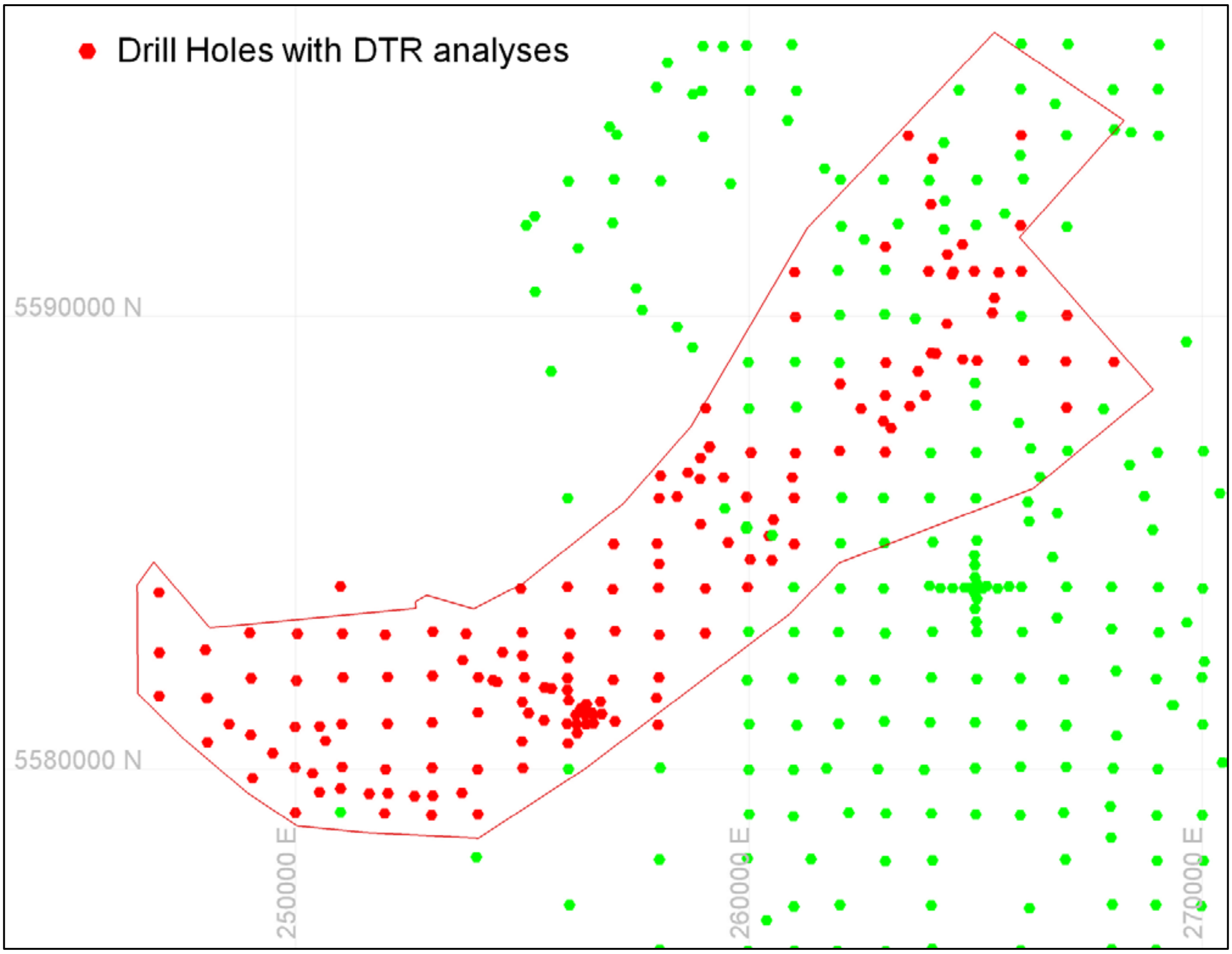


Figure 4-4: DTR Analyses in the Proposed Mining Area

4.3 Dry Bulk Density

Mineral Resource and Ore Reserves, although typically stated in terms of grade and tonnage, are estimated in terms of three parameters: grade, volume and density. Tonnes are the product of volume and density so for good estimation of the resource tonnes a reliable density value must be used for the deposit being evaluated. For a resource estimate the *in situ* dry bulk density is required to estimate the *in situ* tonnage of the deposit.

Using terminology from Lipton 2001, density may be expressed in several ways (Table 4-2) and care should be taken to ensure that the correct density measurement is used.

Table 4-2: Description of Key Density Terms (after Lipton, 2001)

Term	Units	Definition
Specific gravity		Relative density: the ratio of the density of the material to the density of water at 4°C (1.00 tm^{-3})
Density	tm^{-3}	Mass per unit volume
In situ bulk density	tm^{-3}	Density of the material at natural water content
Dry bulk density	tm^{-3}	Density of the material when all water has been dried out of the voids
Grain density	tm^{-3}	Density of the solid grains only – both mass and volume refer to grains only

Density is a measure of mass over volume – the various units used by practitioners are equivalent, e.g. g/cc = t/m^3 = gcm^{-3} = tm^{-3}.

A detailed analysis of the available density data was undertaken previously by Golder in 2010 (Appendix C). From this work the *in situ* bulk density was defined using the Fe regression developed from the calculated theoretical bulk density corrected for measured results (Figure 4-5). The dry bulk density is calculated by the formula ((Fe_2O_3 *0.6994)+81.191)/51.064 where Fe_2O_3 is 69.94% Fe.

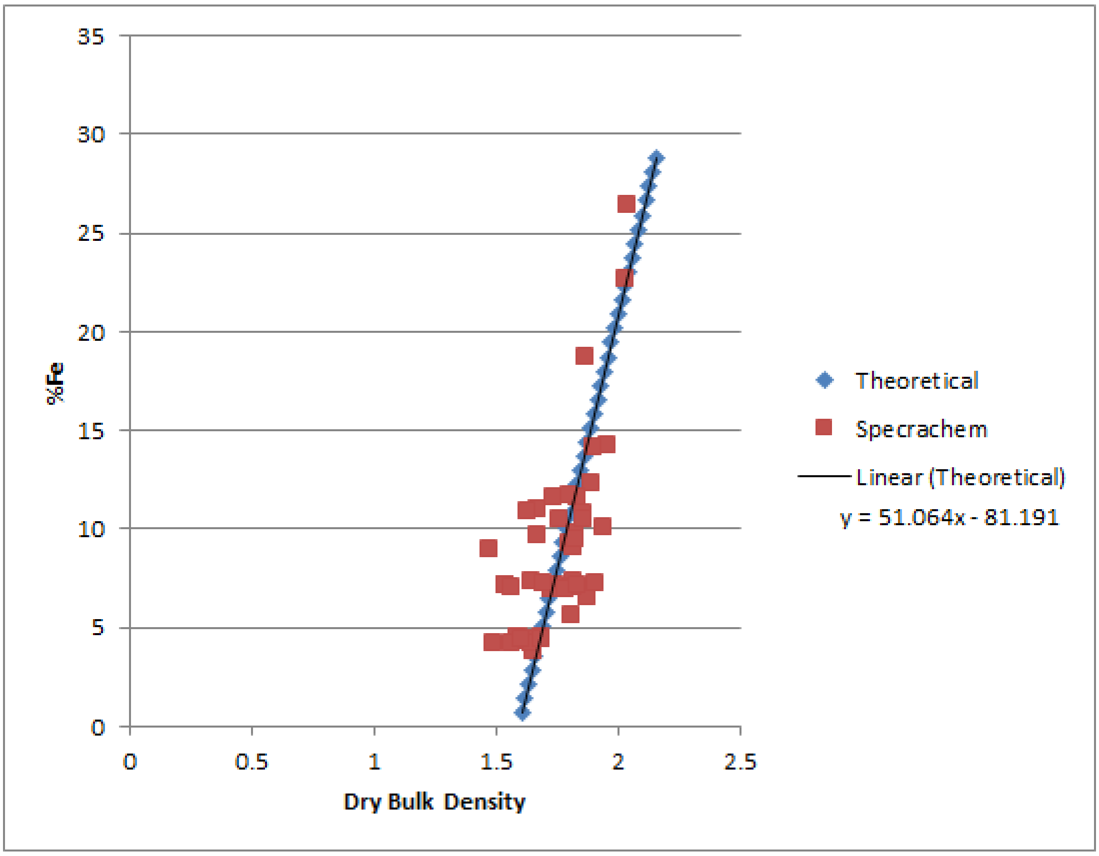


Figure 4-5: Dry Bulk Density Regression against Fe

In 2013 three bulk samples from the pilot plant project were analysed including the determination of the dry bulk density for each. Modelling these results produces a relationship which has higher densities at higher grades (Figure 4-6). This model implies less pore space between the sand particles and is consistent with more small magnetite particles at the higher grades. More particle rounding in the higher grade samples may also be a factor. At the average grade of the deposit in the proposed mine area (7.7 %Fe) the bulk sample model would result in a less than 2% increase in tonnes.

With only three samples supporting this model, and particularly only one sample at the higher grade/higher bulk density end, the model can only be considered indicative and cannot be applied to the resource model.

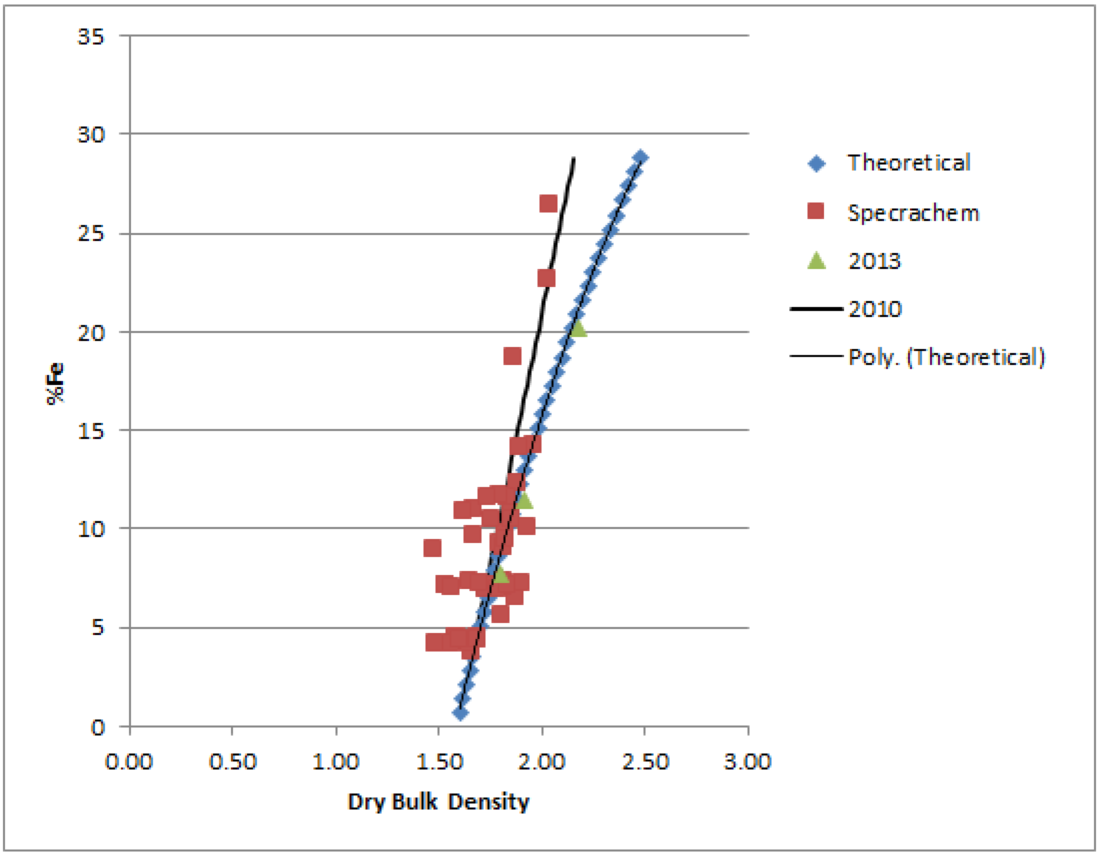


Figure 4-6: Dry Bulk Density Regression against Fe

With consideration of the potential compaction of the sand and minerals other than quartz making up the non-magnetic portion of the sand and the indicative results from the bulk sampling, Golder considers these bulk densities are likely to be slightly conservative.

4.4 Topographic/Bathymetric Surveys

For the South Taranaki project bathymetric data was used to construct a three dimensional model of the sea floor for the area of TTRL's exploration permits.

A variety of data was available over the permit area including nautical charts and sonar readings from miscellaneous sources. All of the available data was used to construct preliminary sea floor models, however a number of anomalous points were present and determining which points were correct was not feasible.

In the proposed mine area updated bathymetric data has been acquired by NZDS using WASSP multibeam sonar data and NIWA using multibeam survey data.

The final sea floor model was constructed by NIWA and a gridded bathymetric model provided to Golder for the 2013 resource model.

4.5 Metallurgical Recovery

In the mineral sand industry the mineralogy and quality can be secondary considerations to the recoverable percentage of heavy mineral. Magnetite and mineral sand deposits are commonly reported with a recovery. For deposits containing magnetically recoverable minerals Davis Tube Recovery (DTR) analysis provides this information. The available DTR analyses now provide recoverable resource figures for the proposed mining area. The pilot plant work, when complete will provide plant recovery and efficiency figures.

In the absence of measured magnetic recoveries, TTRL has in the past referred to titanomagnetite content of the mineral sands as TiFe. TiFe was calculated from the Fe_2O_3 content of the head sample as follows:

$$\textbf{TiFe} = [Fe_2O_3 * 0.6994]*[-2\ mm/100]/0.60$$

Basis:

- All Fe is in titanomagnetite (Metallurgical testing)
- Titanomagnetite contains 60% Fe (assumption)
- Fe_2O_3 contains 69.94% Fe (stoichiometry [2*$Fe^{55.85}$/[2*$Fe^{55.85}$ + 3*O^{16}]])
- The original sample has the +2 mm fraction screened off before sample preparation and analysis so the Fe_2O_3 must be diluted accordingly by multiplying by the -2 mm fraction as a percentage.

5.0 QAQC

5.1 Definitions

- **Quality Assurance** (QA) – is the management system developed to ensure that analytical results are precise and accurate.

- **Quality Control** (QC) – refers to the procedures defined by the QA system designed to measure precision and accuracy.

- **Precision** – is the repeatability of an assay result. It is necessary to know the precision of a set of assays to enable any potential bias to be corrected at a later date. If a batch of assays are repeatable, any bias or accuracy problems can be universally amended. If there is low precision there can be no confidence in the ability to do this.

- **Accuracy** – is a measure of the truthfulness of the assays, i.e. how close they are to reality. Accuracy problems are usually caused by problems with analytical equipment, e.g. XRF calibrations, and can occur at any time. It is important to monitor this on a regular basis – it is much more time dependent than precision and must be corrected as soon as a problem is detected.

- **Bias** – is defined as the distortion in a result or set of results. It can occur as a result of both sampling problems and analytical errors.

- **Certified Reference Materials** (CRM) – are international Iron Ore Standards or standards that have been manufactured locally and have gone through a lengthy and rigorous certification process. Both the mean and the standard deviation of the analyses of the various elements in the sample as determined by the reference laboratories must be known. CRMs are typically inserted by the laboratory into each sample batch as part of their internal QA program. Details of the CRM should be kept in the database as metadata.

- **Internal Reference Materials** (IRM) – are Standards that have been manufactured (usually from discarded pulps) by a certified external laboratory. They have been homogenised and submitted to a series of laboratories in a round robin sufficient to determine the grade distribution of the IRM material. While they have not been certified, both the mean and the standard deviation of the analyses of the various elements in the sample as determined by the laboratory should be recorded in the database as metadata. The standards are submitted blind to the laboratory in sample batches from the field or with pulps for re-assay.

- **Blank** – this comprises either flux or pure silica and is analysed by the laboratory to detect any contamination coming from the sample preparation process or incorrect calibration of analytical equipment. Blanks are used as part of the internal QA procedure for the external laboratories.

5.2 Precision

Precision is the measure of variability or repeatability of the assay results. Precision (or lack of it) is a function of the sample preparation process and the laboratory analytical process.

Precision of the whole chain of sampling, sample preparation and analytical procedures can be quantified by the use of field duplicates, i.e. duplicate field samples that have been split out from the original drill interval in the same manner as the original samples and subjected to exactly the same sample preparation and analytical procedures.

Precision of the analytical procedure can be quantified by duplicate assays on the same sample (repeat analysis of the same pulp).

In this manner, it is often possible to determine where variability occurs in the procedural chain. For example if precision from the whole chain of procedures show low precision, but the precision of both the duplicate assays and the duplicate samples produce high precision, then it can be deduced that the lack of precision originates in the sample splitting.

5.3 Accuracy

The accuracy of the analytical process, the correctness of the result, is quantified on a batch by batch basis by inserting assay standards (either CRM or IRM) and blanks into each batch of samples and monitoring the results over time using a control chart.

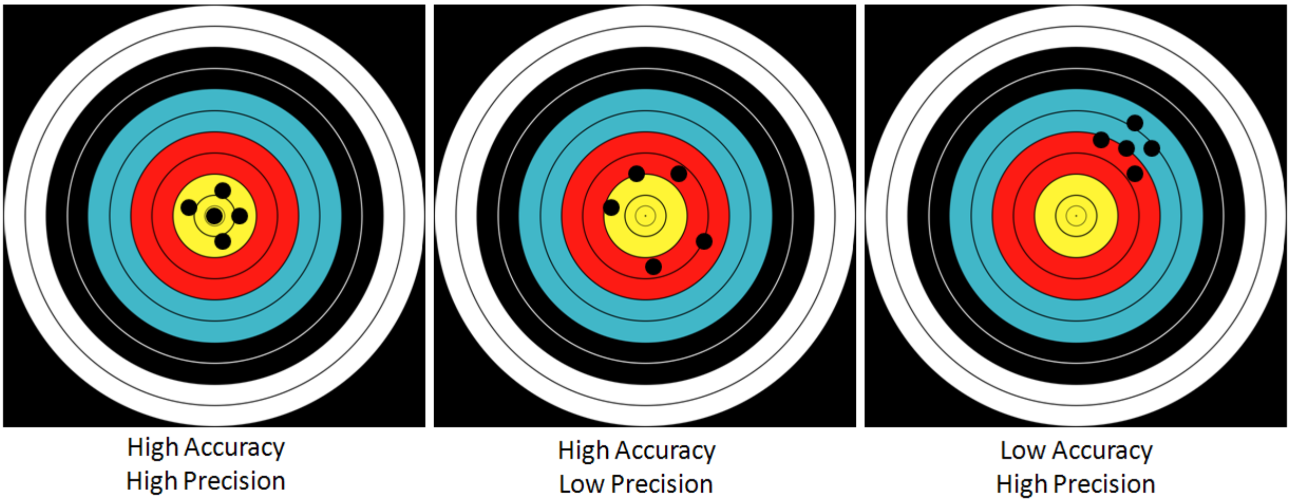


Figure 5-1: Accuracy Control Chart

5.4 South Taranaki

For the initial phase of drilling no QAQC samples were submitted. In order to determine the veracity of the existing analytical results the submission of a series of duplicate samples was recommended. Duplicate samples were resubmitted to the original analysis laboratory, and a second split of the duplicate was sent to Ultratrace as a referee sample. The duplicate samples provide a measure of the laboratory's precision, and the referee samples a measure of their relative accuracy.

Sample submissions since 2011 have included regular submission of blank, duplicate and reference standard samples.

5.4.1 Data

QAQC data was received from TTRL digitally in the same access database as the drill hole data – *TTR_DB1_To_Golders_20130903.accdb* as table *tbl_QAQC*. The database contains results for 708 QAQC samples. The identification number for each sample includes a code identifying the type of QAQC sample. Table 5-1 summarises these.

Table 5-1: QAQC Sample Identification and Numbers

Type	Code	Code Position	Description	Number
Duplicate	A, B, C, D, X	Suffix	Spectrachem (CRL) field duplicate	244
	X	Suffix	18 SGS referee samples	18
	X2, R., Rpt	Suffix	Ultratrace and SGS Perth referee samples	207
			Subtotal	***469***
Standard	STD	Suffix or Prefix	TTRL Standard Sample	108
	STD	Prefix	Concentrate – unused	8
			Subtotal	***116***
Blank	BLK	Prefix	TTRL Blank Sample	6
	BK2, BK3	Suffix	TTRL Blank Sample	117
			Subtotal	***123***
			Total	**700**

5.4.2 Blank Samples

A blank sample is normally comprised of 100% silica sources from crushed quartz or glass bottles.
The blank samples originally being submitted by TTRL were from a beach sand with significant contaminants and only 80% SiO_2 (Table 5-2). The 2% Fe_2O_3 value for the standard is quite significant when compared to the average mineralisation grade of around 9% Fe_2O_3. The standards are useable but contamination would be more obvious if a 'pure' silica sample were used. Golder's recommendation of the use of a commercial quartz gravel as the blank sample was adopted part way through the drilling program (Figure 5-2).

Table 5-2: Preliminary Blank Sample Analysis

Analyte	Assay
Fe_2O_3	1.96
MnO	0.03
TiO_2	0.23
CaO	1.97
K_2O	1.66
P_2O_5	0.07
SiO_2	79.67
Al_2O_3	8.91
MgO	0.75
Na_2O	2.55
LOI	1.80
Total	99.60

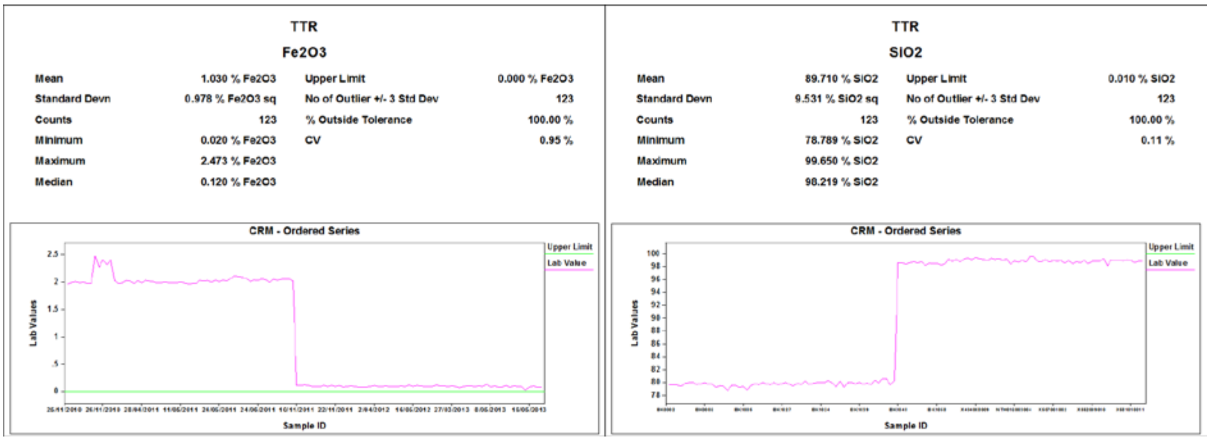


Figure 5-2: Blank Sample Results (Sequenced by Date)

Results

The blank samples show no evidence of contamination during sample preparation and analysis.

The use of high silica blanks can impact on the calibration of XRF equipment. It should be confirmed with the laboratory that there is no issue with the submission of high silica blank samples.

Plots for all analytes are included in Appendix D.

5.4.3 Standard Samples

TTRL have produced a standard sample (IRM) from a blend of iron sand samples taken during drilling. Thirty samples of the standard blend were analysed. These analyses were collated and the average grade and standard deviation for each analyte set as the expected result. The analyses of standard samples submitted with drill samples are compared to the expected result. Figure 5-2 and Table 5-3 illustrate the standard sample analysis results for the major analytes representing 99.5% of the total sample.

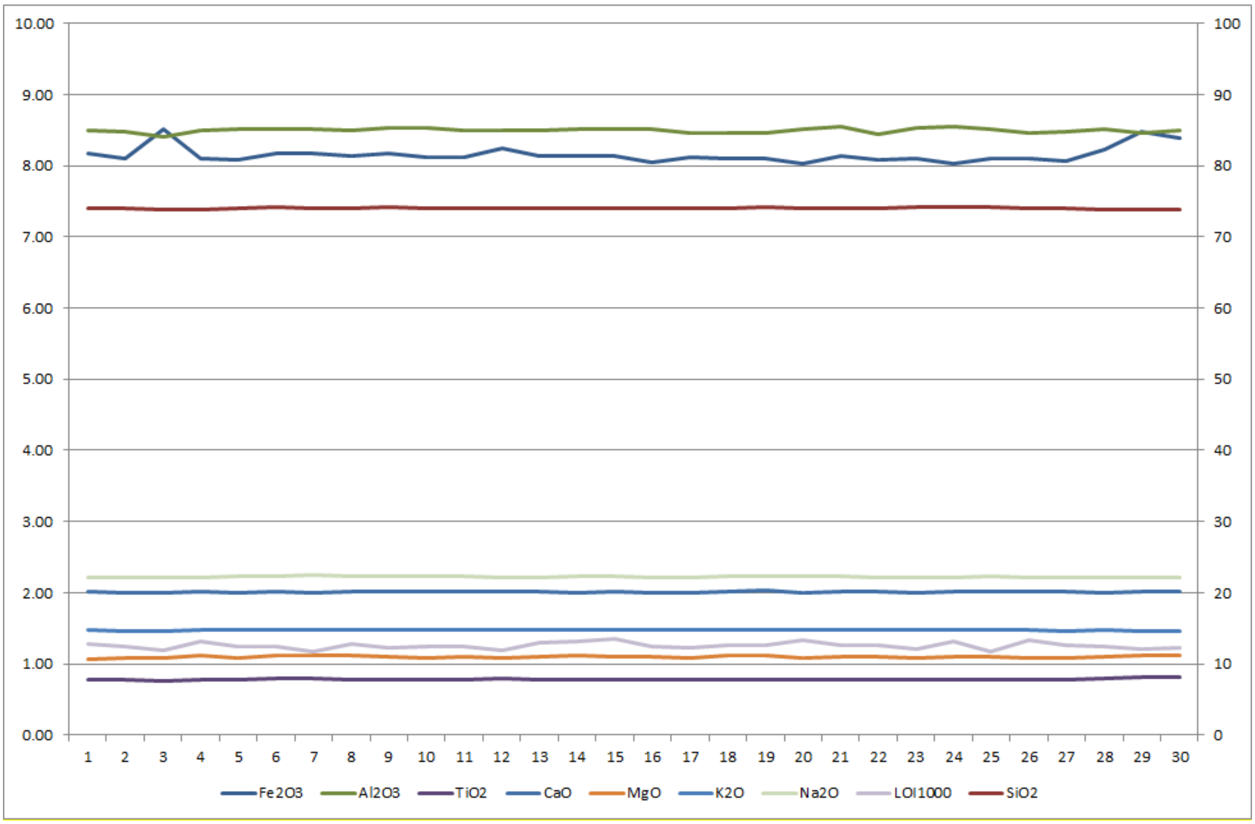


Figure 5-3: Standard Analysis Data – Major Analytes

Table 5-3: Major Analytes – Standard Analysis

	Fe_2O_3	SiO_2	Al_2O_3	TiO_2	CaO	MgO	K_2O	Na_2O	LOI^{1000}
Standard Deviation	0.12	0.10	0.03	0.01	0.01	0.01	0.01	0.01	0.05
Average	8.15	74.02	8.50	0.78	2.01	1.10	1.47	2.22	1.26
Minimum	8.02	73.80	8.40	0.76	2.00	1.07	1.46	2.21	1.17
Maximum	8.51	74.17	8.56	0.81	2.03	1.11	1.48	2.24	1.36

Results

The standard shows variable results. Although showing more erratic behaviour in early analyses most analytes are now returning results within acceptable ranges. The exceptions and anomalies are noted below.

Na_2O shows a change in average result part way through the drilling program suggesting a recalibration or sample preparation change that should be investigated (Figure 5-6). The shift in value is consistent so a correction may be possible when the cause of the problem is demined. At the same time that the Na_2O values shifted a number of other analytes show subtle changes in the returned results as well.

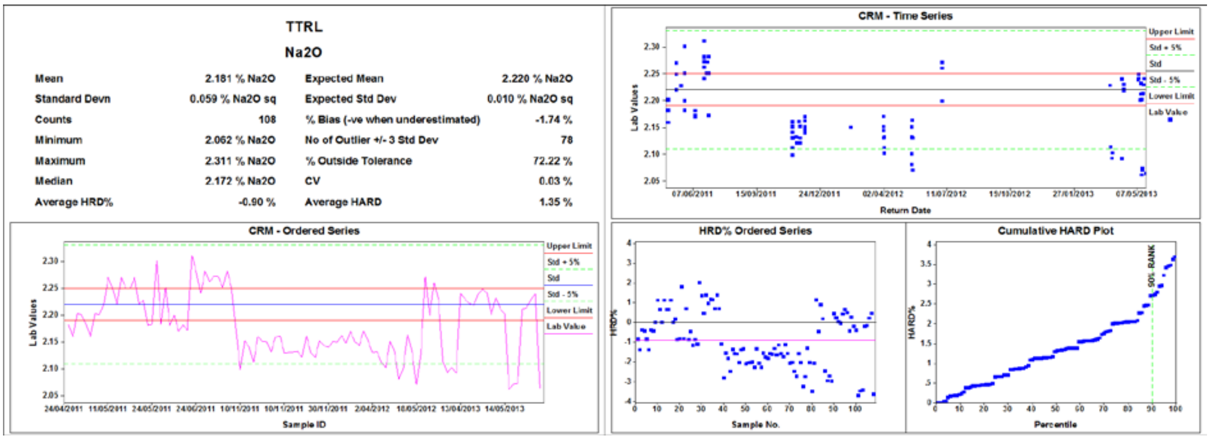


Figure 5-4: Na_2O Standard Analysis Results

P_2O_5 is returning a slightly high result compared to the original standard analysis. Part of the problem appears to be in the equipment precision at the low values being reported. The standard has a value of 0.084% but the laboratory is only returning results in 0.01% increments. MnO shows a similar problem.

CaO is returning slightly high but acceptable values. The issue here may by with the original analysis.

Al_2O_3 is returning lower values during the last drilling campaign (Figure 5-5). The values are still within acceptable limits but future results should be monitored for any further drift away from the expected value.

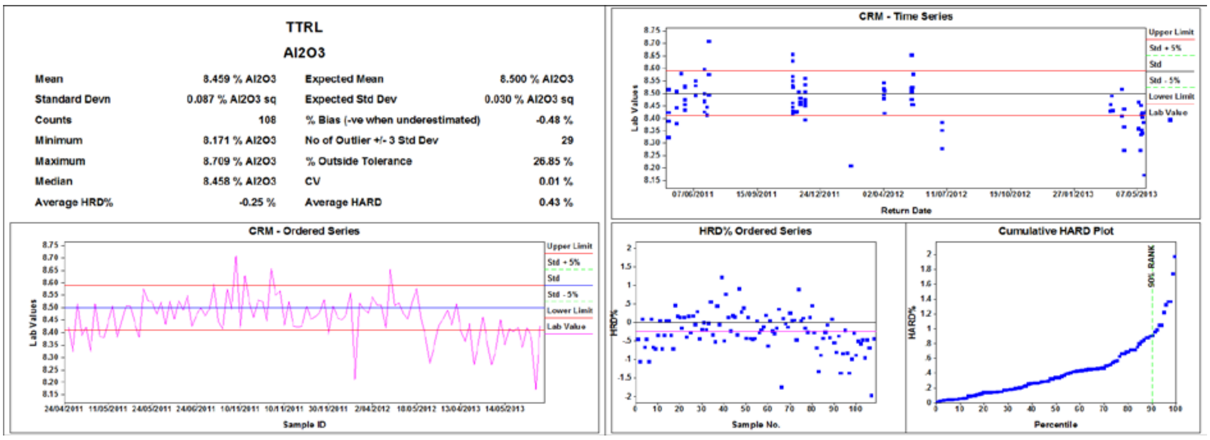


Figure 5-5: Al_2O_3 Standard Analysis Results

Plots for all analytes are included in Appendix D.

5.4.4 Duplicate Samples

The duplicate samples available for analysis are of three types:

1) Field Duplicates – head samples
2) Referee Samples – head grades
3) Referee Samples – Concentrate grades.

Field Duplicates

The field duplicate samples are a second sample taken from the original drill returns and analysed at the same laboratory as the primary sample. The primary and duplicate samples have both been sent to the Spectrachem Analytical in Lower Hutt. Spectrachem have been part of CRL Energy Ltd since 2008. The sample preparation and analytical technique are the same for both samples. Comparison of the paired samples indicates the repeatability or precision of the results.

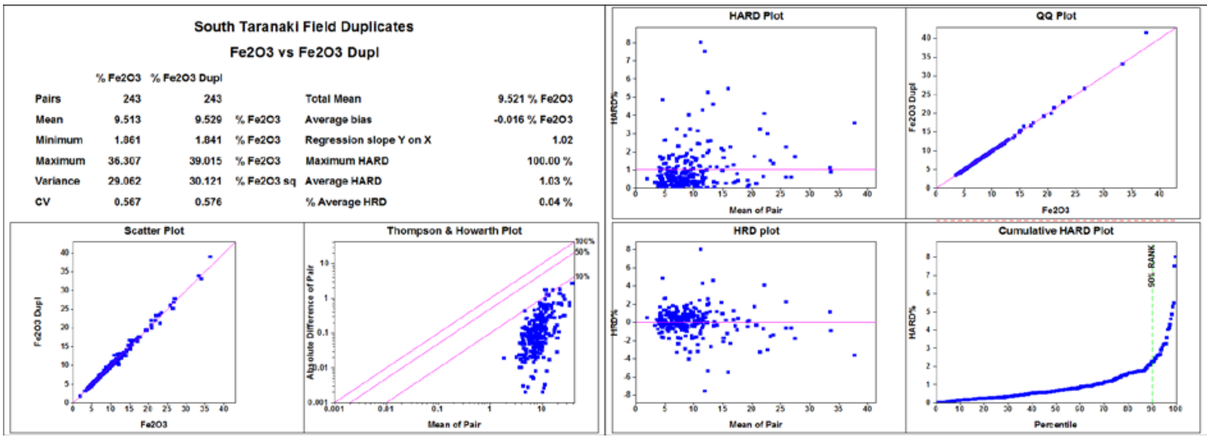


Figure 5-6: Duplicate Analysis – Fe_2O_3

Duplicate samples show good correlation between the original and duplicate sample for most analytes. Figure 5-6 illustrates the comparison of results for Fe_2O_3. P_2O_5 is the least repeatable result with the duplicate sample biased slightly high (Figure 5-7). The low values and equipment precision may be the cause.

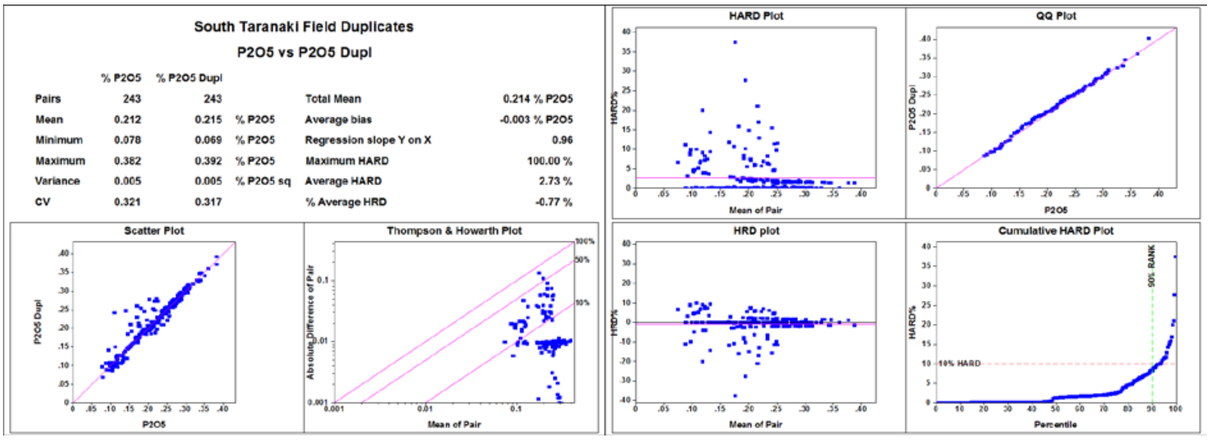


Figure 5-7: Duplicate Analysis – P_2O_5

Plots for all analytes are appended.

Referee Samples – Head Grade

In the absence of QAQC data for the pre 2011 drilling programs a selection of samples were sent to SGS (Auckland?) and Ultratrace (Perth).

Most analytes show a good correlation between the original and duplicate analyses. Some analytes are showing minor differences at higher grades. These are not considered material at present but should be monitored. P2O5 and Na2O show the least repeatable results being consistently overestimated and underestimated by 10% and 6% respectively (Figure 5-8,Figure 5-9). The referee samples sent to SGS show a similar trend for P_2O_5 (Figure 5-10), however the Na_2O results show much better correlation with Spectrachem.

Plots for all analytes are included in Appendix D.

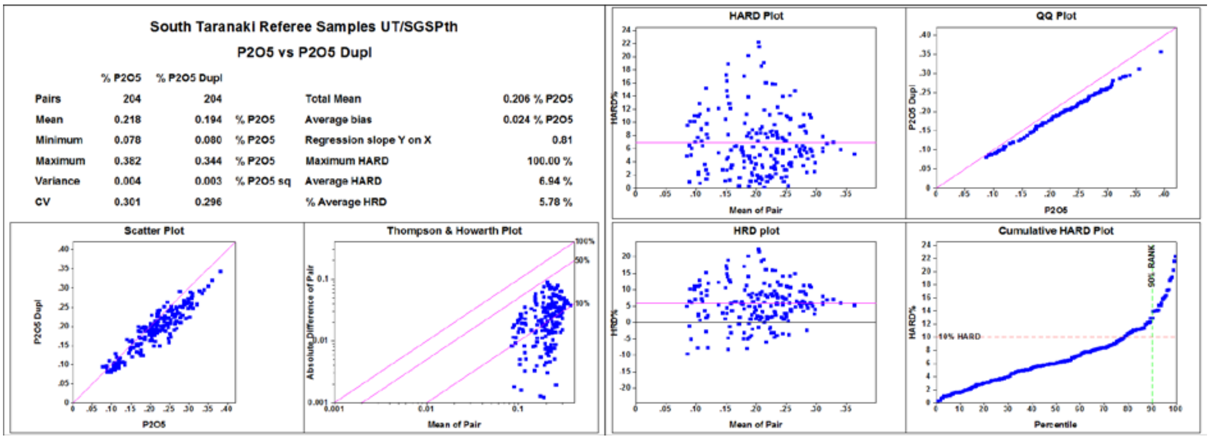


Figure 5-8: Referee Sampling – P_2O_5

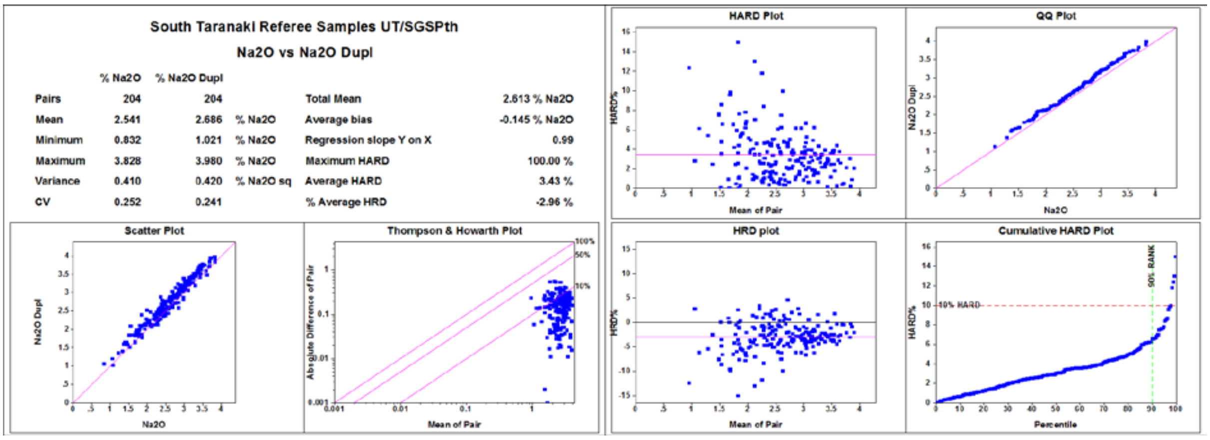


Figure 5-9: Referee Sampling – Na_2O

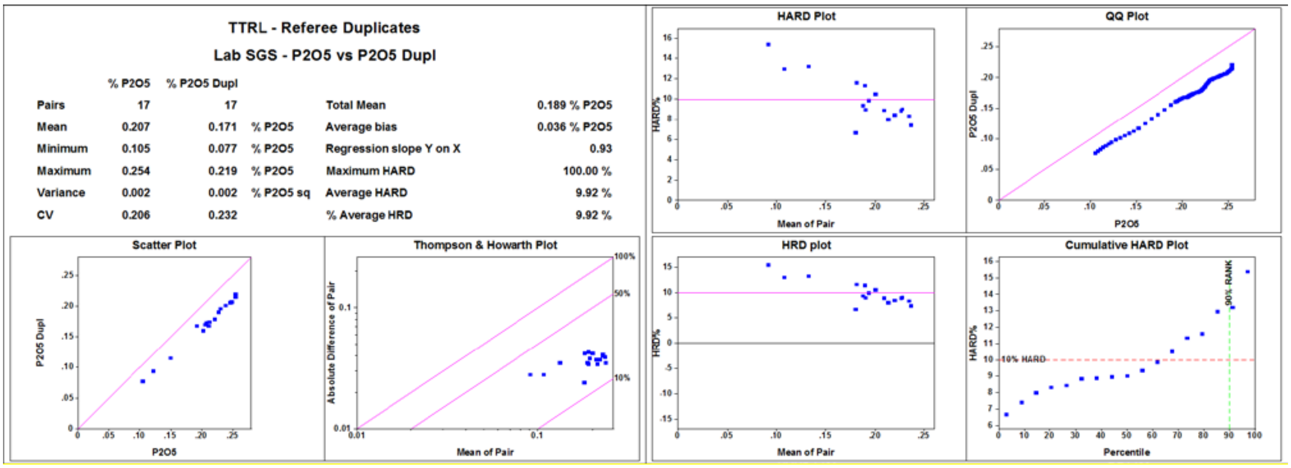


Figure 5-10: Referee Head Sample Results – SGS – P_2O_5

Referee Samples – Concentrate Grade

26 samples have been submitted to Amdel for Davis Tube Recovery determination and concentrate analyses. A head Fe analysis was also supplied for each sample. Overall the main analytes show a good correlation. The minor analytes with low concentrations show variable correlations due to what appear to be detection limit differences between the laboratories. The DTR and head Fe grades show very good correlation. Concentrate Fe and Ti slow slight positive biases of 2% and 5% respectively. Na_2O and SiO_2 (Figure 5-11) shows more variance in sample correlation, most likely a product of the effectiveness of the DTR process in removing the gangue minerals.

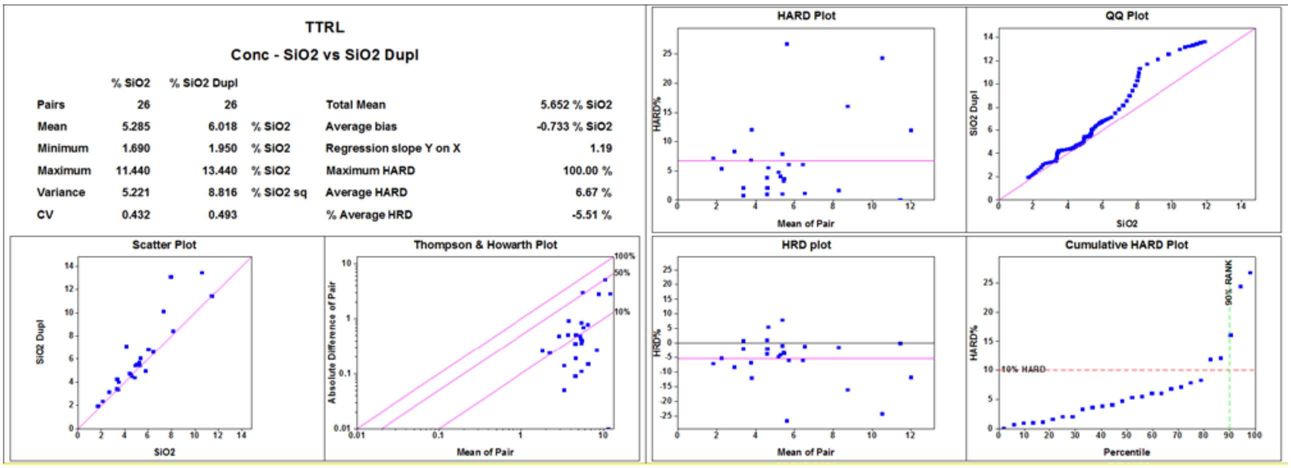


Figure 5-11: Referee Concentrate Sample Results – SGS – SiO_2

Plots for all analytes are included in Appendix D.

5.4.5 Independent Laboratory Review

Jeremy Batchelor of Chem Tek Consulting undertook an independent lab audit and QAQC data analysis in 2013 finding the laboratory procedures and results satisfactory (*Technical Review of Data and Methodology used by CRL with respects to the TTRL sample program, 17/06/2013*).

5.4.6 Conclusions and Recommendations

With consideration of the comments made in this review, most of the concerns previously raised by Golder regarding the quality of the analytical data for the South Taranaki deposit have been alleviated with the increased volume of QAQC samples available.

Standards

The use of high silica blanks can impact on the calibration of XRF equipment. It should be confirmed with the laboratory that there is no issue with the submission of high silica blank samples.

Head Samples

The QAQC sampling confirms the accuracy and precision of the head grade analytical results is adequate. Current trend in the standard sample results for Al_2O_3 needs to be monitored to ensure the accuracy of the analyses is maintained. The standards sample results for Na_2O indicate a change in the analytical technique, possibly in the instrument calibration.

Analyses for P_2O_5 should be checked. Spectrachem appear to be overestimating grades when compared to the referee laboratories.

Concentrate Samples

Golder understands a standard sample for submission with concentrate samples is being sourced for inclusion with future sample submissions. Duplicate sampling at referee laboratories shows that the results are independently repeatable and indicates the accuracy of samples is good. A standard sample will confirm the accuracy of the results.

Golder considers the analytical results suitable for inclusion in the South Taranaki resource estimate.

6.0 GEOLOGY MODELLING

6.1 Regional Geology

The New Zealand region lies in the southwest of the Pacific Ocean astride a distinct belt of volcanic and earthquake activity that surrounds the Pacific Ocean (Figure 6-1). This is the Pacific Mobile Belt or "Ring of Fire" and the activity results from the structure of the Earth's crust. The crust is made up of a number of segments called plates, which move relative to one another in response to forces deep within the Earth. The plates may rub past one another, one may be forced down below another, or they may buckle at the edges as they meet head on. Wherever there is a plate boundary there is geological activity of a volcanic or tectonic nature. New Zealand straddles the boundary between the Pacific and Indian-Australian plates. To the north of New Zealand and beneath the eastern North Island, the thin, dense, Pacific plate moves down beneath the thicker, lighter Indian-Australian plate in a process known as subduction; within the South Island the plate margin is marked by the Alpine Fault and here the plates rub past each other horizontally; while south of New Zealand the Indian-Australian plate is forced below the Pacific plate. Plate movement results in volcanic activity in the North Island and in earthquakes that are felt throughout the country (GNS, 2010).

Figure 6-1: New Zealand Regional Tectonic Setting

6.2 Local Geology

The ironsand deposits being investigated are all currently located below sea level. The accumulations of ironsands are geomorphological features interpreted as littoral and channel deposits, i.e. Dunes, Beaches, Deltas and River Channels

The source of the titanomagnetite making up the potentially economic part of the deposit is the andesitic volcanoes of the Taranaki and Central Volcanic regions. Andesites contain 3-4% titanomagnetite which after surviving the weathering of the host rock is transported to the coast by rivers and streams and then south by littoral drift (Lecointre, 2006).

The titanomagnetite and other heavy minerals are naturally concentrated by wave action so beach and strandline accumulations are common.

Sea level is not constant over geological time. During marine regression, when the sea level drops, new littoral, or shoreline, zones are defined and rivers incise the now exposed marine terraces.

Marine transgression over the last 10 000 years has seen the sea level change by up to 30 m in response to eustatic changes (Figure 6-2). When the sea level rises the incised channels are drowned and filled with sediment and a new littoral zone with associated beach and dune deposits is developed. When the sea level is stable for a period there is the more prolonged development of beaches, dunes and marine terraces and their associated ironsand deposits. These deposits were formed approximately 7 000 and 9 000 years before present (BP) (Figure 6-2).

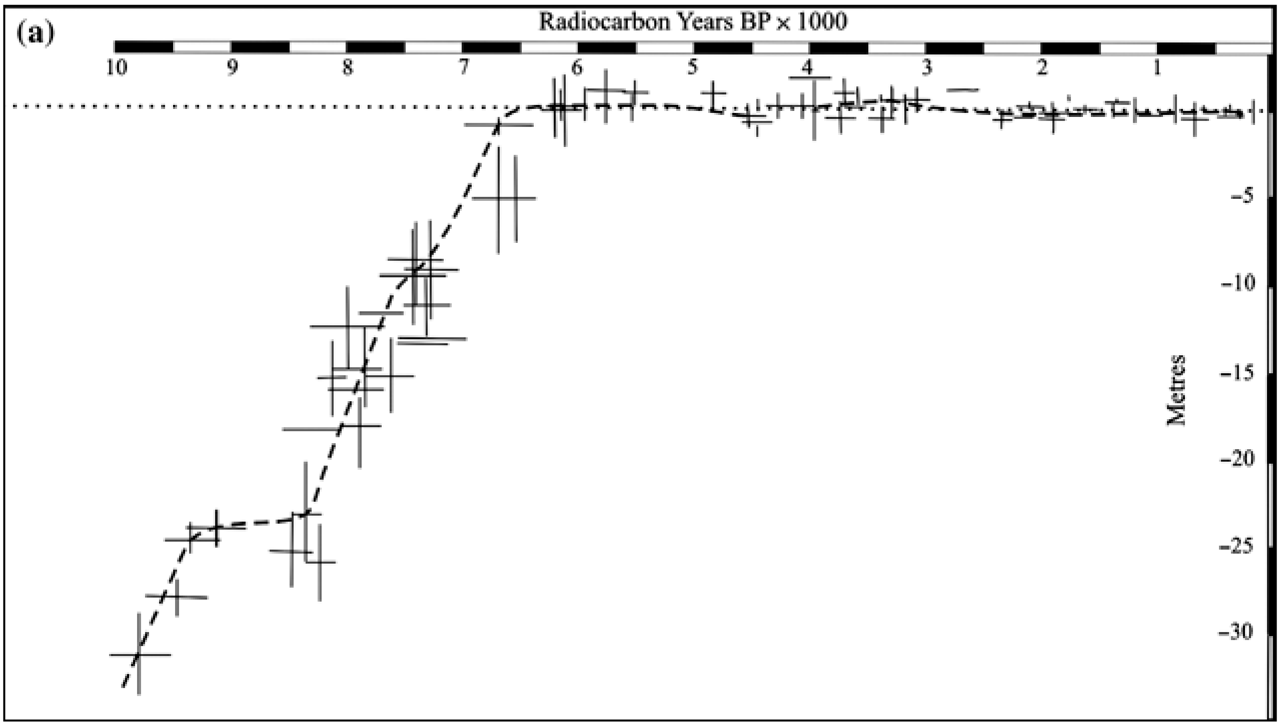


Figure 6-2: Sea Level Change (Gibb, 1986)

The titanomagnetite has been further concentrated by fluvial and aeolian action reworking the sediments over the last 6000 years.

Owing to the strong magnetic properties of the titanomagnetite in the ironsands, magnetic geophysical surveys over the area clearly show the palaeo features where the mineral has accumulated (Figure 6-3). The old river channels can clearly be seen as extensions of the current on shore drainage system. The 10 m and 25 m bathymetric contours coincide with the 7000 and 9000 year BP standstills and are associated with distinct deposits sub-parallel to the palaeo shorelines. The 70 m and 120 m bathymetric contours coincide with earlier marine stand stills at 14 000 and 20 000 years BP.

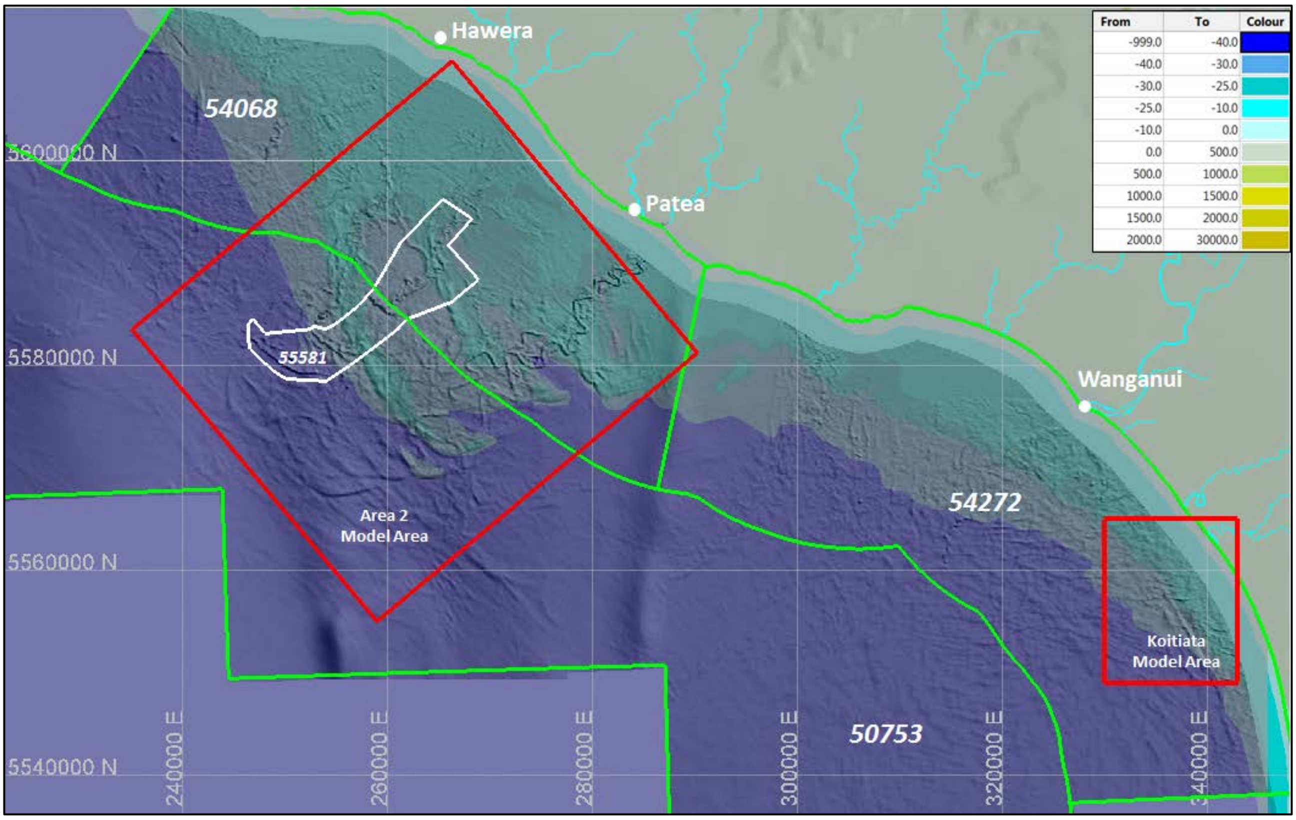


Figure 6-3: Geophysical Image (Magnetics) Overlaying Bathymetry

Figure 6-4 and Figure 6-5 illustrate ironsand accumulations at the Patea River mouth and Waipipi Beach respectively. Most of the Taranaki shoreline contains black sands with varying titanomagnetite content.

Figure 6-4: Patea River Mouth

Figure 6-5: Waipipi Beach

6.3 Mineralisation

The South Taranaki ironsand deposits are comprised principally of silica sand with minor dark green clinopyroxene, black orthopyroxenes, hornblende and titanomagnetite (Orpin, 2010). In addition to the sands the samples commonly contain up to 15% shells and pebbles. Titanomagnetite is the only magnetic mineral recorded to date.

The mineralogy and chemical analysis suggest that most of the Fe content of the sands is in the titanomagnetite. FeO, Fe_2O_3 and TiO_2 are only available for a limited number of samples. Plotting the FeO:Fe_2O_3:TiO_2 ratios identifies the mineral species as a titanium enriched magnetite (Figure 6-6).

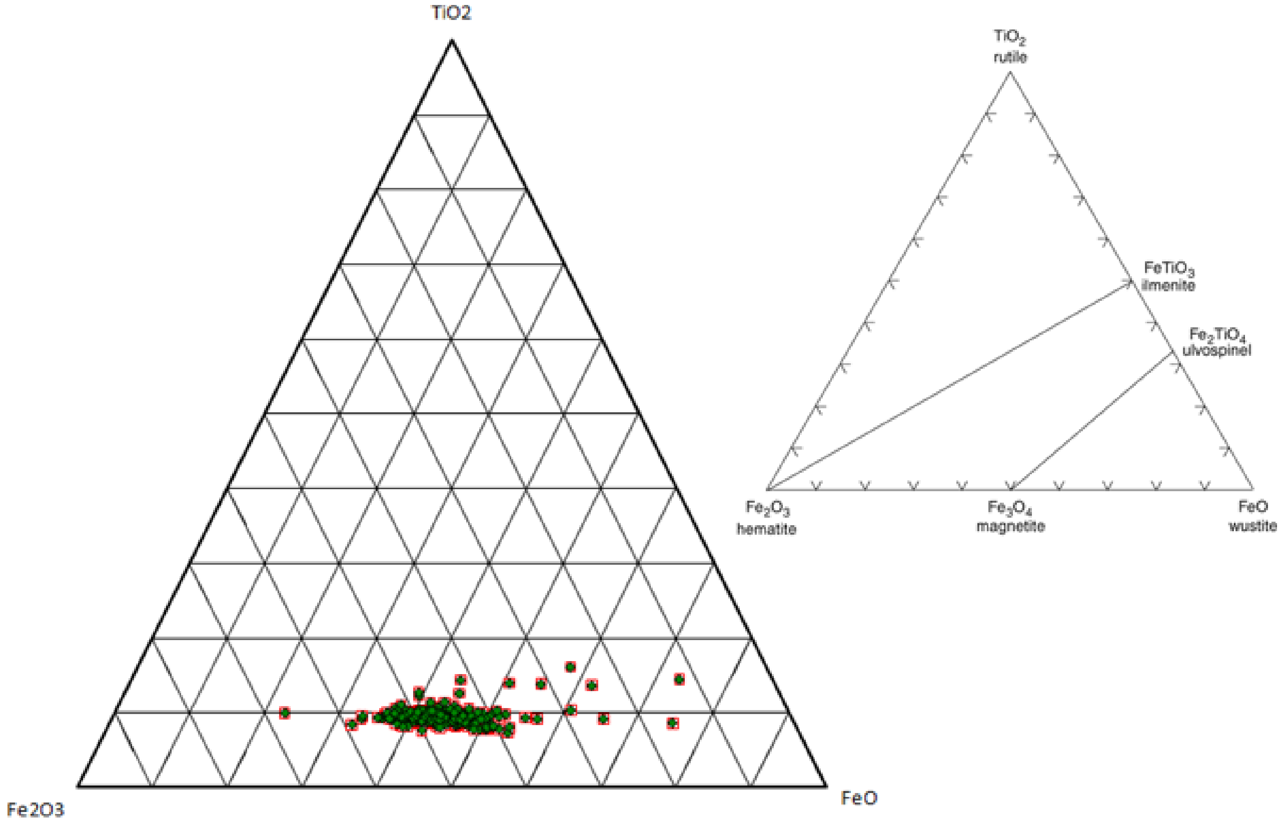


Figure 6-6: Fe_2O_3-FeO-TiO_2 Ternary Plot

6.4 Geological Model

The original geological model used to target drilling assumed higher grade material would be intersected where the geophysics showed a higher magnetic response.

Statistical and visual analysis of the drill hole sample data up to 2012 showed that the samples were relatively consistent across most locations with only a small high grade population (Figure 6-7). This conflicted with the anticipated result of getting higher grade samples where the geophysical survey showed higher magnetic values as illustrated in Figure 6-8 A.

The geological model was revised to include a layer of overburden covering the features seen in the geophysical survey imagery. A blanket of reworked sands, as illustrated in Figure 6-8 B, would explain the relatively consistent results from the shallow drilling.

The infill drilling over the proposed area in 2013 has shown the blanket of sand to be up to 30 m thick. The closer spaced drill data has also shown there is lateral variation in grades consistent with the geophysical data. Figure 6-9 illustrates the comparison between the magnetic geophysical data and full hole composite grades for the drilling in the proposed mine area. A mathematical relationship between the geophysical data and the drill sample data has not been investigated beyond the 2011 work that found no direct correlation.

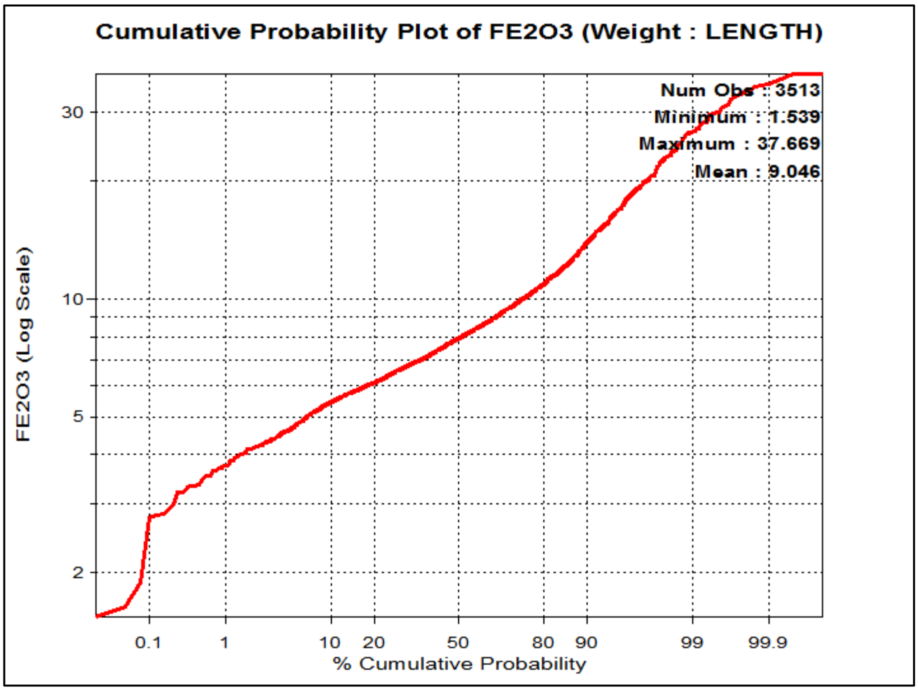


Figure 6-7: Fe_2O_3 – All Drill Holes

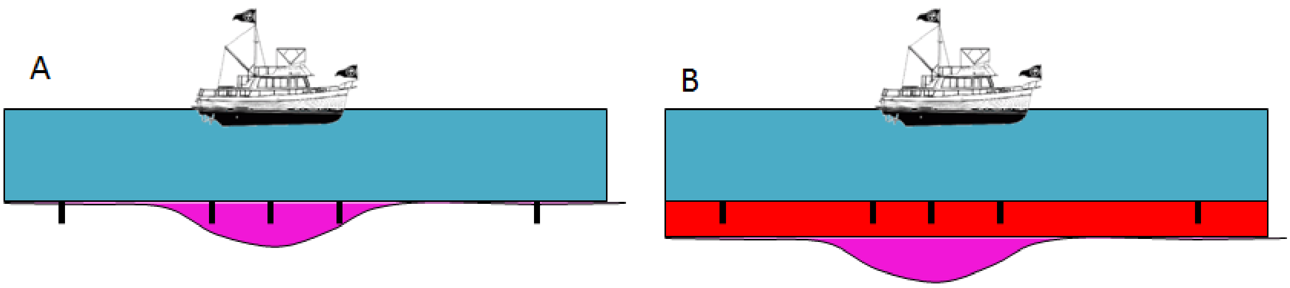


Figure 6-8: Revised Geological Model

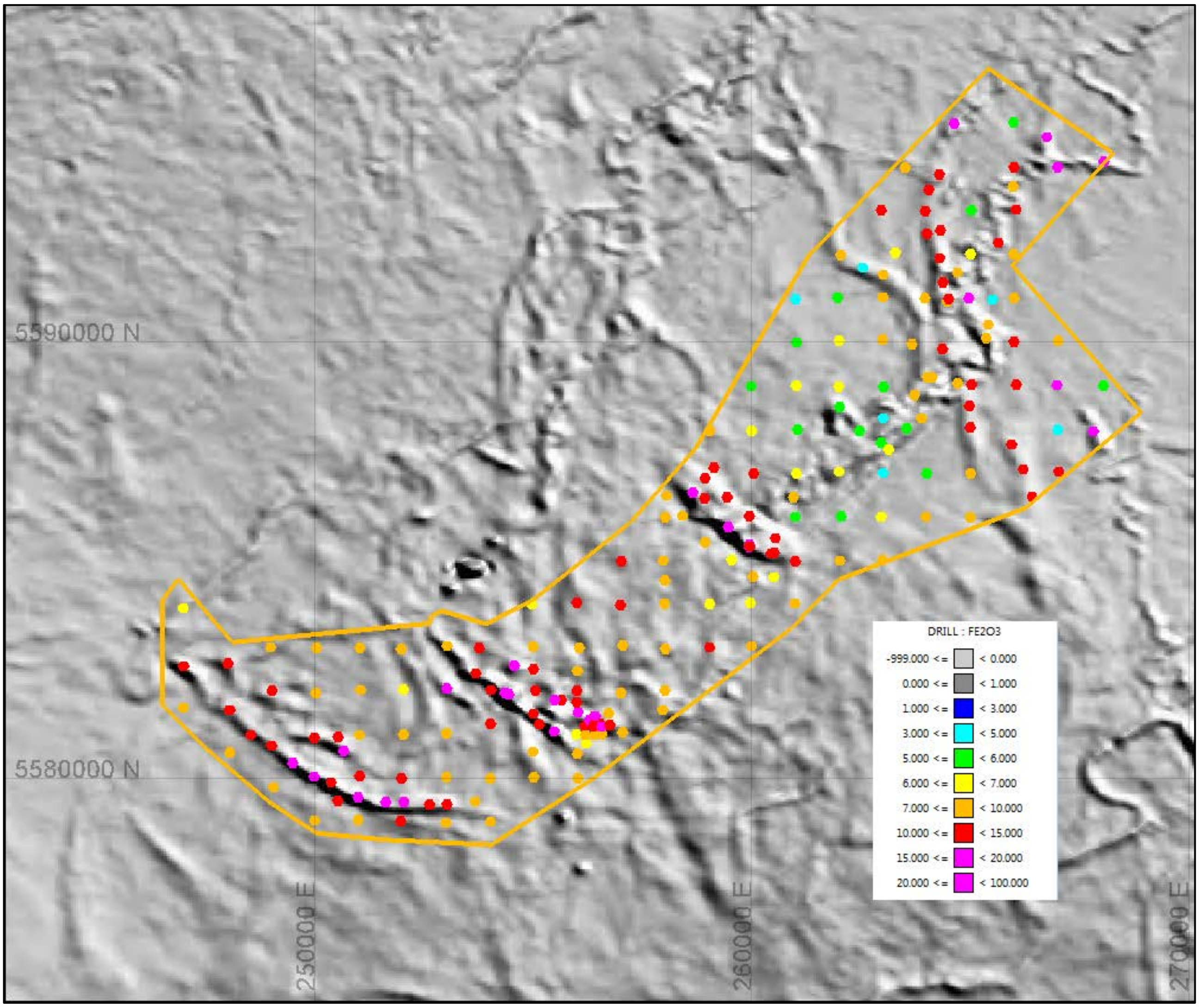


Figure 6-9: Magnetic Geophysical Response Compared to Drill Sample Analyses

The current geology model is the sand blanket with the base defined by the depth of drilling. The true thickness of the deposit has not been defined by drilling over most of the permit area. The limited deeper drilling and seismic profiling indicate thicknesses approximately 30 m to 40 m. Laterally the deposit is divided into geomorphological domains as discussed in Section 6.5.

6.5 Geological Domains

6.5.1 Principle of Domaining

In order to achieve meaningful statistical studies and a robust estimate, the sample data should be domained into geologically homogenous areas. Domains in which the statistical properties of the population do not change is the concept of stationarity.

6.5.2 Domains

The geological model has defined an overburden layer of sand which is different to the underlying geomorphological features. However, these overburden sands have been reworked from the material making up these underlying features. Based on this, a series of broad domains were defined over the area sampled by the drilling. These are illustrated in Figure 6-10. The old river channels are defined as fluvial zones, Graham Banks is defined as dunes and the linear features further off shore in Domain 9 are interpreted as slumps. The remaining northern areas are defined as deltas and Koitiata as a palaeo beach.

The domains were further refined to limit the extent of the influence of any particular drill hole to approximately 1000 m horizontally. This was done in order to stop an unreasonable volume of material receiving an estimated grade in the block model. The 1000 m extrapolation is based on the drill spacing of 2000 m required for an Inferred Resource in this deposit (Appendix E).

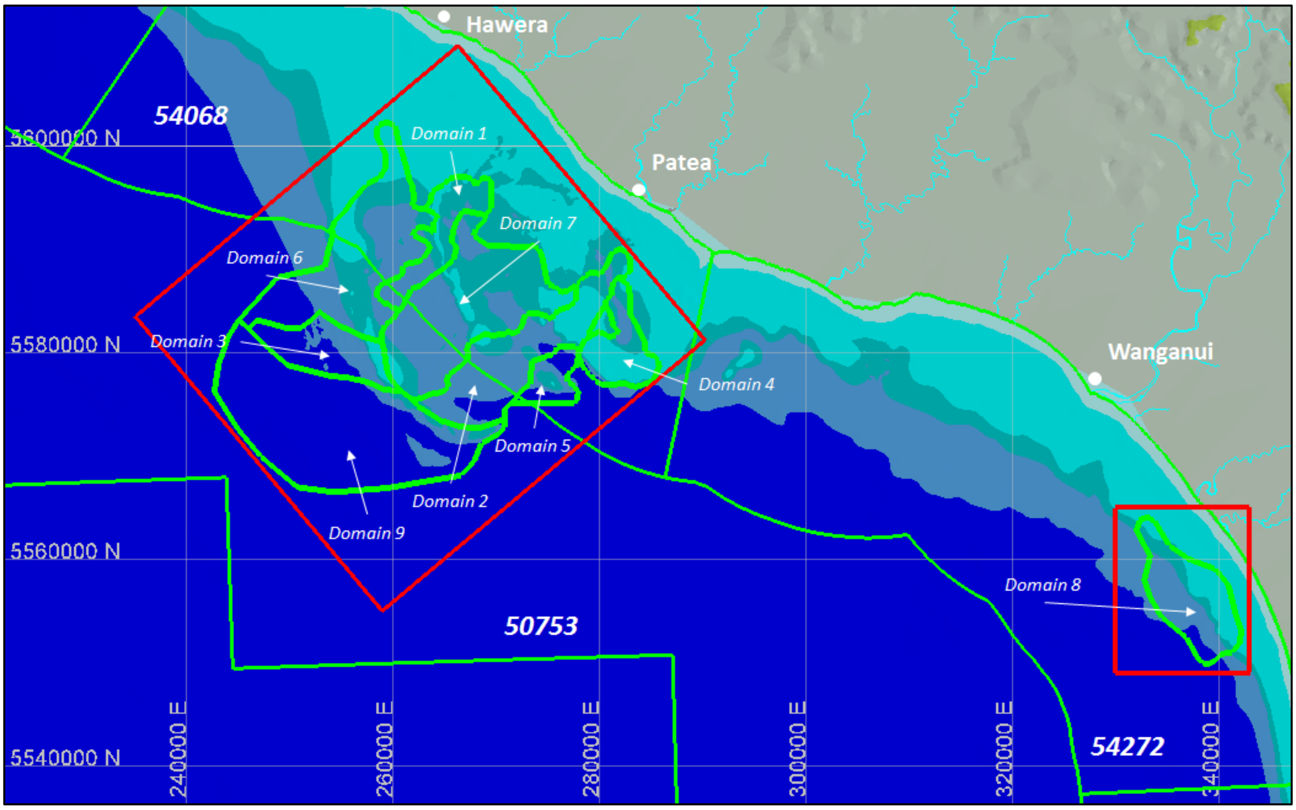


Figure 6-10: Domains

Figure 6-11 illustrates the cumulative log probability plots of Fe_2o_3 head assays for domains in the Area 2 deposit and shows that there are statistical differences between the domains supporting the approach taken. Koitiata (Domain 8) is geographically separated from the Area 2 domains.

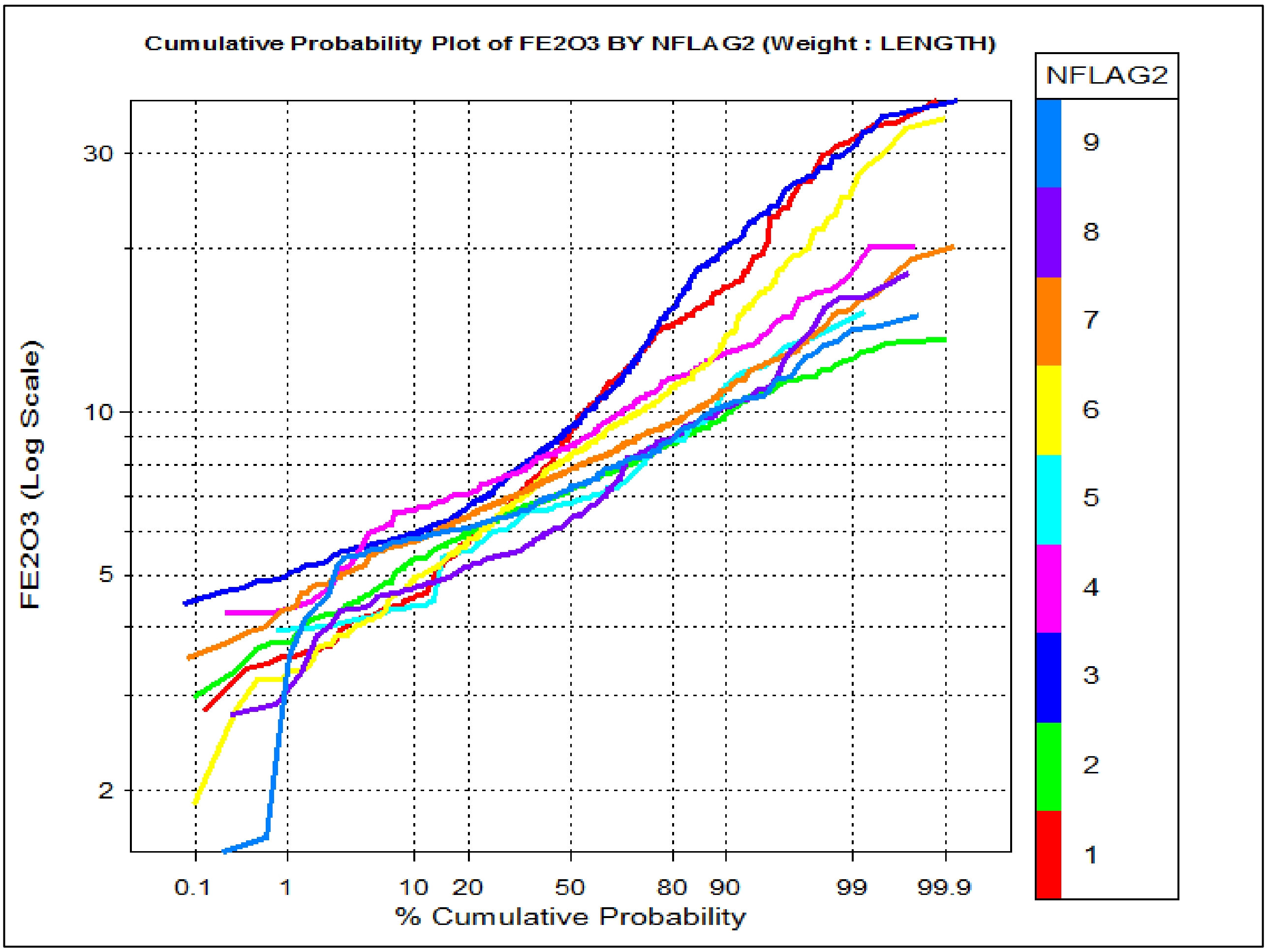


Figure 6-11: Cumulative Probability for Fe_20_3 by Domains for the Area 2 and Koitiata Deposits.

In addition to the geomorphological (spatial) domains, a mineralised zone was applied where all samples greater than or equal to 4% Fe_2O_3 were included in the mineralised zone. The change in the population at 4% can be seen in Figure 6-7. There are also higher other natural breaks in the population at 5.5% Fe_2O_3 and 11% Fe_2O_3. These upper breaks have not been investigated further to date. To define the lower boundary of the mineralisation an intersection selection method was used to generate composites of the drill hole sample database using a 4% target with a maximum of 2 m internal waste. As the proposed mining method of dredging will not be removing waste separately, overburden was blended into the selection. Multiple intersections were manually assessed to determine where to define the base of mineralisation by either incorporating the subgrade material or raising the base of mineralisation.

7.0 GEOLOGICAL BLOCK MODEL

Block modelling is carried out so that interpolation techniques can be used to estimate grades for points between the known data points, within geologically defined volumes. In order to increase the resolution of geological boundaries within the block model a process called sub-blocking is adopted, whereby the blocks are split into sub-blocks at wireframe boundaries. Sub-blocks are re-aggregated wherever possible to produce intermediate-sized sub-blocks within a parent block, thus reducing the size of the model for computational purposes. Figure 7-1 shows the location of the two block models constructed. The domains outside the model areas contained insufficient samples to produce a robust estimate.

Note that for this resource update the Area 2 model has been rotated to align the model blocks with the general direction of mineralisation and the proposed mining direction.

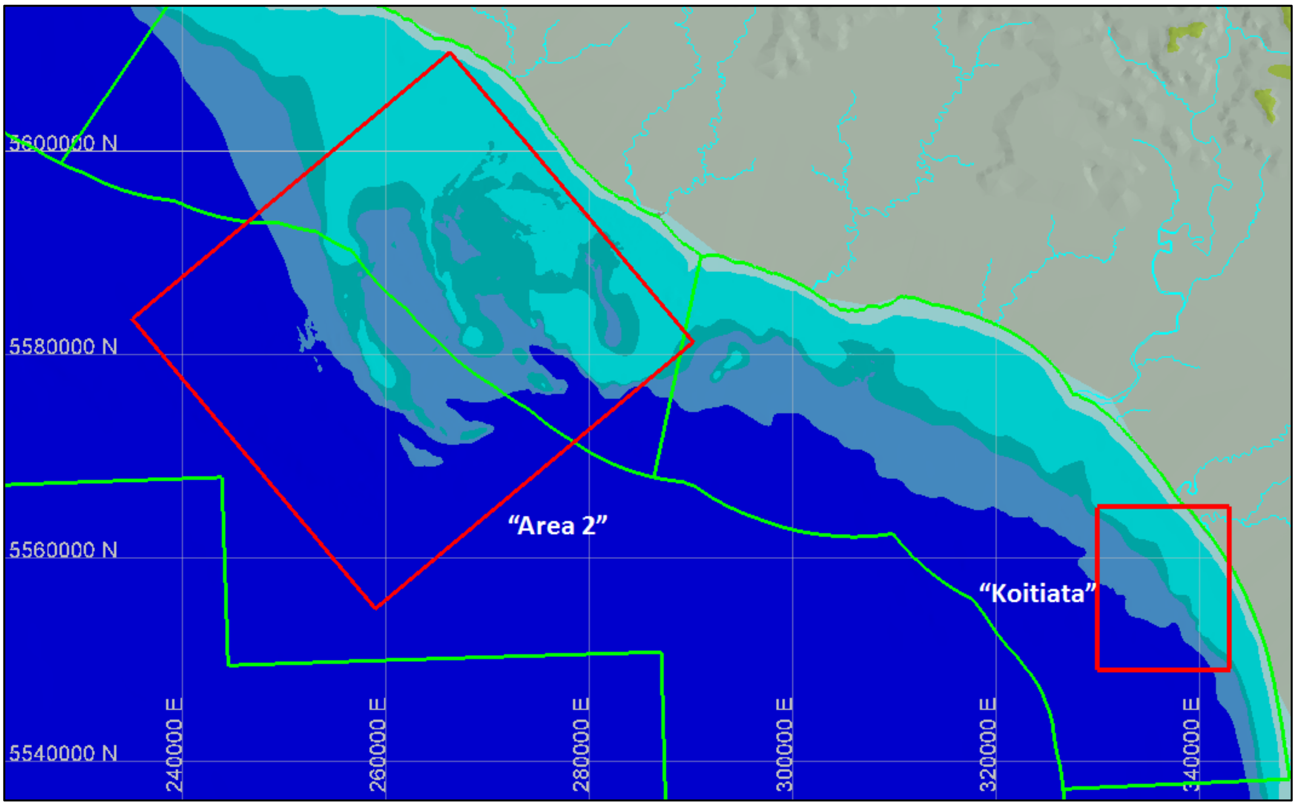


Figure 7-1: Block Model Boundaries

7.1 Block Model Parameters

Table 7-1 details the position and dimensions of the block models and the dimensions of the blocks. These are illustrated in Figure 7-1.

Table 7-1: Block Model Origins and Extents (m)

	Area 2		Koitiata	
	Parent Block	**Sub-Block**	**Parent Block**	**Sub-Block**
X Min	259000	259000	330000	330000
Y Min	5555000	5555000	5549000	5549000
Z Min	-110	-110	-110	-110
Bearing	50	50	0	0
Dip	0	0	0	0
Plunge	0	0	0	0
X Extent (m)	40800	40800	13000	13000
Y Extent (m)	37200	37200	16000	16000
Z Extent (m)	110	110	110	110
X Size (m)	300	50	500	50
Y Size (m)	300	50	500	50
Z Size (m)	1	1	1	1

Table 7-2 details the block model variables, data type and default value for the Area 2 and Koitiata block models. Head grades are for the recovered (-2 mm) sand. Concentrate grades are for the magnetically recoverable portion of the recovered sand. Head grades are weighted by sample recovery in the variables "*acc_**". Concentrate grades are weighted by sample recovery and magnetic recovery (DTR) in the variable "*acc_dt_**".

Table 7-2: Block Model Variables

Variables	Default	Type	Description
horizon	-9	float	Mineralisation Zone
zone	-9	float	Domain
mine	-9	float	Mine Area Flag
sg	-9	float	Bulk Density
fe2o3	-9	float	Fe_2O_3 Head Grade
acc_fe2o3	-9	float	Fe_2O_3 Accumulation
rec_fe2o3	-9	float	Fe_2O_3 Recovery Weight
acc_head_pass	-9	float	Estimation pass
acc_head_samp	-9	float	Number of samples used
acc_head_hole	-9	float	Number of holes used
acc_head_dist	-9	float	Average Distance to samples
acc_head_slope	-9	float	Slope of regression
al2o3	-9	float	Al_2O_3 Head Grade
acc_al2o3	-9	float	Al_2O_3 Accumulation
rec_al2o3	-9	float	Al_2O_3 Recovery Weight
loi	-9	float	LOI Head Grade
acc_loi	-9	float	LOI Accumulation

Variables	Default	Type	Description
rec_loi	-9	float	LOI Recovery Weight
p2o5	-9	float	P_2O_5 Head Grade
acc_p2o5	-9	float	P_2O_5 Accumulation
rec_p2o5	-9	float	P_2O_5 Recovery Weight
sio2	-9	float	SiO_2 Head Grade
acc_sio2	-9	float	SiO_2 Accumulation
rec_sio2	-9	float	SIO_2 Recovery Weight
tio2	-9	float	TiO_2 Head Grade
acc_tio2	-9	float	TiO_2 Accumulation
rec_tio2	-9	float	TiO_2 Recovery Weight
cao	-9	float	CaO Head Grade
acc_cao	-9	float	CaO Accumulation
rec_cao	-9	float	Cao Recovery Weight
k2o	-9	float	K_2O Head Grade
acc_k2o	-9	float	K_2O Accumulation
rec_k2o	-9	float	K_2O Recovery Weight
mgo	-9	float	MgO Head Grade
acc_mgo	-9	float	MgO Accumulation
rec_mgo	-9	float	MgO Recovery Weight
mno	-9	float	MnO Head Grade
acc_mno	-9	float	MnO Accumulation
rec_mno	-9	float	MnO Recovery Weight
rec	-9	float	REC Head Grade
dt_fe	-9	float	Fe_2O_3 Concentrate Grade
acc_dt_fe2o3	-9	float	Fe_2O_3 Concentrate Accumulation
rec_dt_fe2o3	-9	float	Fe_2O_3 Concentrate Weight
acc_dt_pass	-9	float	Estimation pass
acc_dt_samp	-9	float	Number of samples used
acc_dt_hole	-9	float	Number of holes used
acc_dt_dist	-9	float	Average Distance to samples
acc_dt_slope	-9	float	Slope of regression
dt_al2o3	-9	float	AL_2O_3 Concentrate Grade
acc_dt_al2o3	-9	float	Al_2O_3 Conc Accumulation
rec_dt_al2o3	-9	float	Al_2O_3 Conc. Weight
dt_loi	-9	float	LOI Concentrate Grade
acc_dt_loi	-9	float	LOI Conc Accumulation
rec_dt_loi	-9	float	LOI Weight
dt_p	-9	float	P_2O_5 Concentrate Grade
acc_dt_p2o5	-9	float	P_2O_5 Conc Accumulation
rec_dt_p2o5	-9	float	P_2O_5 Conc. Weight
dt_sio2	-9	float	SiO_2 Concentrate Grade
acc_dt_sio2	-9	float	SiO_2 Conc Accumulation

Variables	Default	Type	Description
rec_dt_sio2	-9	float	SiO_2 Conc. Weight
dt_tio2	-9	float	TiO_2 Concentrate Grade
acc_dt_tio2	-9	float	TiO_2 Conc Accumulation
rec_dt_tio2	-9	float	TiO_2 Conc. Weight
dt_cao	-9	float	CaO Concentrate Grade
acc_dt_cao	-9	float	CaO Conc Accumulation
rec_dt_cao	-9	float	CaO Conc. Weight
dt_k2o	-9	float	K_2O Concentrate Grade
acc_dt_k2o	-9	float	K_2O Conc Accumulation
rec_dt_k2o	-9	float	K_2O Conc. Weight
dt_mgo	-9	float	MgO Concentrate Grade
acc_dt_mgo	-9	float	MgO Conc Accumulation
rec_dt_mgo	-9	float	MgO Conc. Weight
dt_mn	-9	float	MnO Concentrate Grade
acc_dt_mno	-9	float	MnO Conc Accumulation
rec_dt_mno	-9	float	Mno Conc. Weight
acc_dtr	-9	float	DTR Accumulation
dtr	-9	float	DTR Head Grade
acc_dtr_est	-9	float	DTR_EST Accumulation
dtr_est	-9	float	DTR_EST Estimation
acc_dt_est_samp	-9	float	DTR_EST number of samples
acc_dt_est_hole	-9	float	DTR_EST number of holes
acc_dt_est_pass	-9	float	DTR_EST estimation pass
class	-9	float	Resource Classification
sg_rec	-9	float	SG Accumulation
mag_fe	-99	float	magnetic fe
fe_yield	-99	float	Fe Yield
dtr_reg	0	integer	0 = estimated dtr; 1 = dtr by regression

7.2 Model Domain Codes

Table 7-3 lists the wireframe models used to code the block models for Koitiata and Area 2.

The horizon variable refers to the mineralised zone and is defined by the bathymetric surface at the top and a lower surface defined by a 4% Fe_2O_3 cut-off. Minor subgrade material was absorbed at the top of the mineralisation horizon to accommodate the mining model (dredging) which will not be able to separate overburden. The lower surface was constructed from drill hole points which were modelled as a surface. This surface was then folded to run sub-parallel to the bathymetric surface. The zone variable is coded by the geomorphological domain.

Table 7-3: Block Model Boundary Coding

Wireframe	Variable	Code	Priority	Inversion	Projection
gol_bathy_topo_2011.00t	horizon	1	1	None	Along Z Axis
A2_base_of_minz_folded.00t	horizon	0	2	None	Along Z Axis
gol_bathy_topo_2011.00t	horizon	-99	3	Partial	Along Z Axis
Domain_001.00t	zone	1	1	None	Along Z Axis
Domain_002.00t	zone	2	2	None	Along Z Axis
Domain_003.00t	zone	3	3	None	Along Z Axis
Domain_004.00t	zone	4	4	None	Along Z Axis
Domain_005.00t	zone	5	5	None	Along Z Axis
Domain_006.00t	zone	6	6	None	Along Z Axis
Domain_007.00t	zone	7	7	None	Along Z Axis
Domain_008.00t	zone	8	8	None	Along Z Axis
Domain_009.00t	zone	9	9	None	Along Z Axis
gol_bathy_topo_2011.00t	zone	-99	10	Partial	Along Z Axis
A2_base_of_minz_folded.00t	zone	0	11	None	Along Z Axis

Owing to the large size of the model all blocks above or below the mineralised horizon and all of those outside the modelled zones were deleted from the model to facilitate the ease of its manipulation and end use.

7.3 Geological Block Model Validation

The block model was sliced on regularly spaced intervals in section and plan and compared against modelled wireframes. The model was verified for:

- Prioritisation of wireframes during the block construction
- Blocks correctly flagged when compared with wireframes
- Any unassigned blocks within the block model, and
- Volumetric/geometric consistency of coded blocks against wireframe models.

Examination of the block model revealed no inconsistencies from the intended plan. Particular attention was paid to ensuring that the vertical sub-block size was adequate to model the thin mineralisation horizon.

8.0 STATISTICAL ANALYSIS

8.1 Compositing

Samples are normally composited to a length longer than the raw sample length. Moving to a longer support length reduces the variance, reflecting the reduction in spikes in the assay data and reducing the impact of isolated outlier values. The composite length is chosen to minimise differences in support for all samples and moderate the sample variance whilst retaining sufficient numbers of samples for grade estimation.

The chosen length is commonly a compromise between smoothing of the original variability and maintaining sufficient resolution for the geological wireframes. The South Taranaki data was composited to 1 m.

Less than 4% of raw samples are less than 1 m long. Two samples are two metres long. (Figure 8-1).

Assessment of scatter diagrams between small sample length and various assays shows a poor correlation which is not expected to have a material impact on the final estimates (e.g. Fe_2O_3 in Figure 8-2). To manage samples less than 1 m long, length weighted samples were used for statistical analysis and estimation.

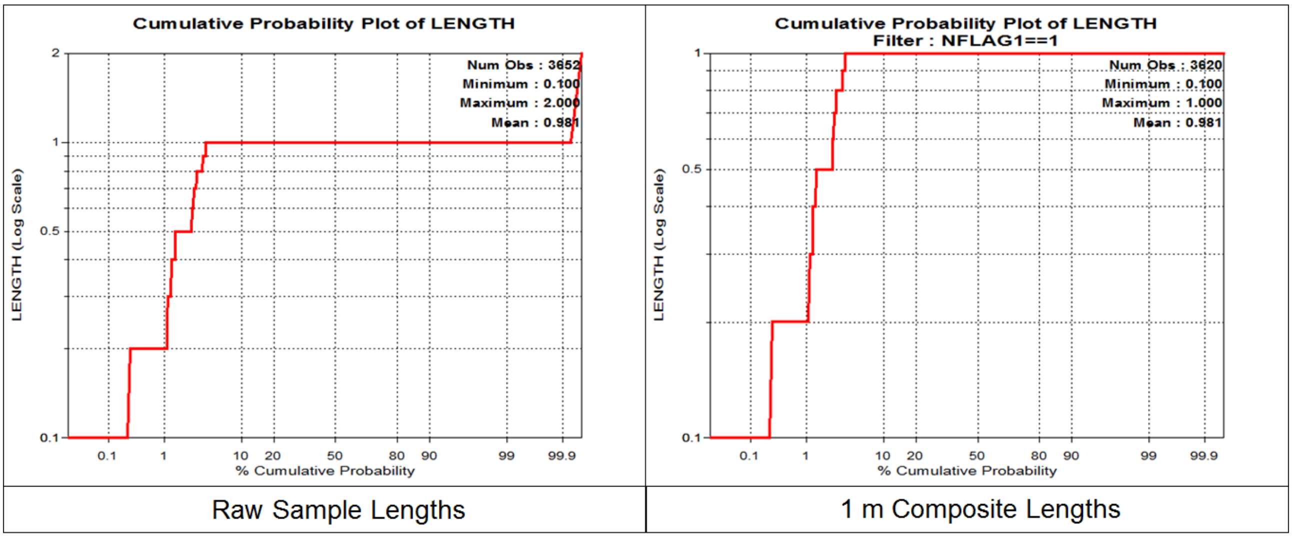


Figure 8-1: Cumulative Probability Plots – Sample and Composite Lengths

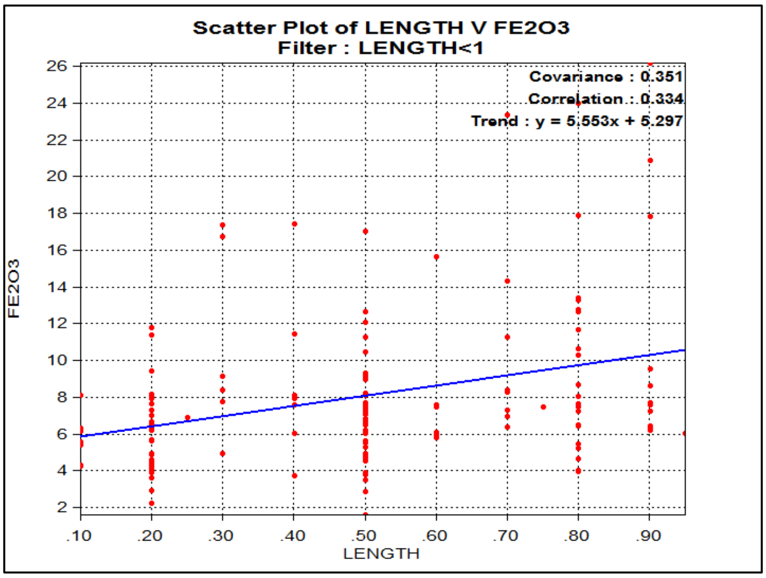


Figure 8-2: Scatter Plot – Short Samples vs Fe_2O_3

8.2 Data Flagging

Statistical studies are most meaningful when dealing with data from geologically homogenous populations. To allocate the data most effectively to the various spatially defined geological populations, the raw sample data are intersected with the wire-framed triangulations and flagged according to the parts of the geological interpretation to which they relate.

For the Taranaki South deposit sample flagging of the raw data used the same coding scheme as the block model (zone code in Table 7-3). Compositing was carried out on the flagged sample dataset broken by Ore Horizon and recording the majority code for the Domain.

8.3 Univariate Statistics

Exploratory data analysis (EDA) was carried out on the 1 m composite data and involved descriptive univariate and scatter diagram analysis and distribution comparisons. Distribution comparisons were also carried out to assess the differences between estimation groups.

Table 8-1 summarises the statistics for the main analytes by mineralised domain. Table 8-2 provides a statistical summary for the Davis Tube concentrate results. It should be noted that the Davis Tube analysis has only been carried out for a limited data set located in the proposed mining area (Figure 4-3). Davis Tube data extensively covers the Domain 3 (i.e. the highest Fe_2O_3 head grade) and partially occupies Domain 1 and Domain 6 with some minor amounts in Domain 7 and Domain 9. As per Figure 8-3,Domains 3, 1 and 6 contain the highest Fe_2O_3 and DTR values.

Inspection of Figure 8-3 shows that while a number of domains are statistically similar for the Fe_2O_3 head grade they appear distinctively differently when other elements, such as SiO_2 and Al_2O_3, are considered.

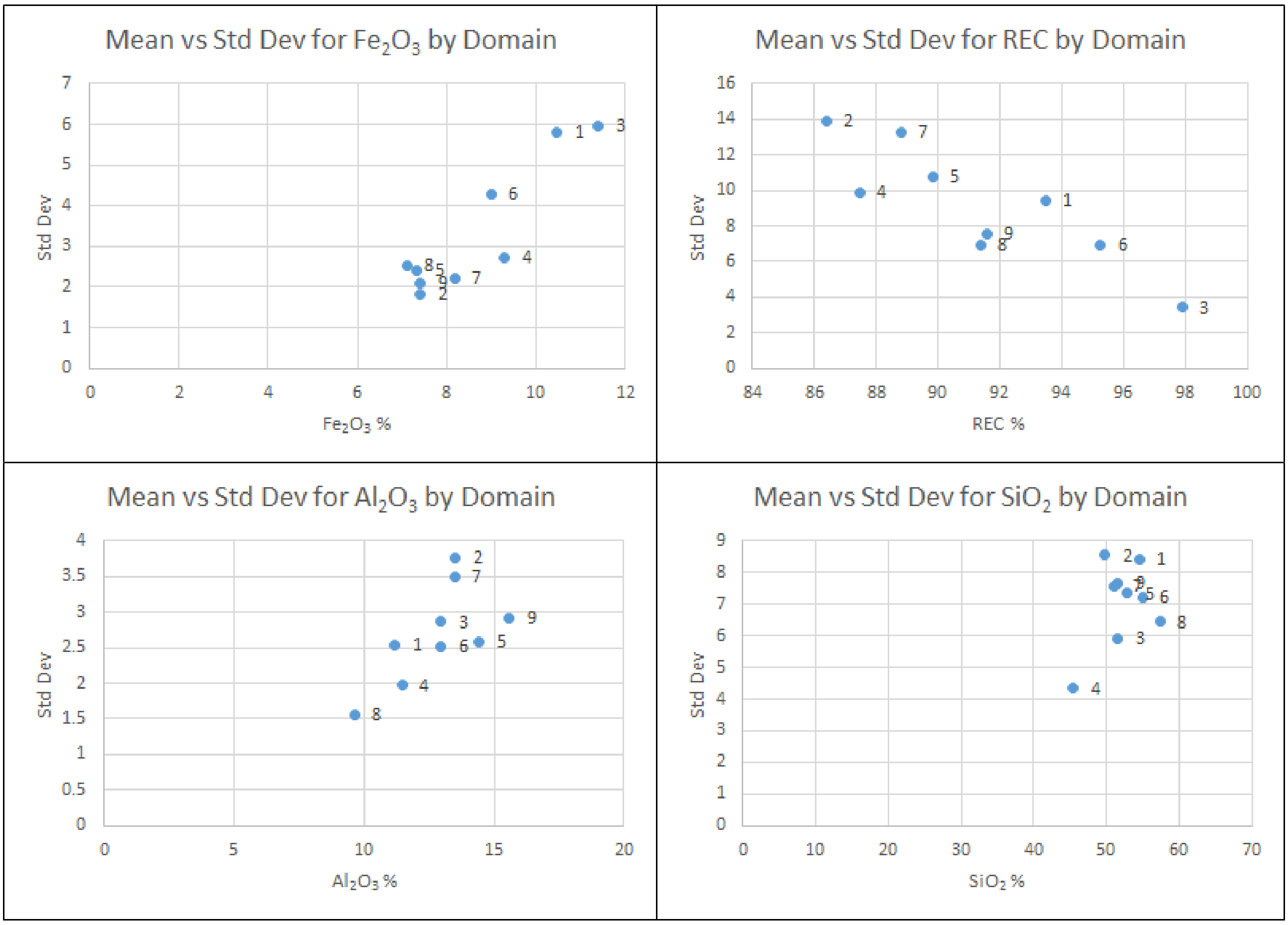


Figure 8-3: Mean vs Standard Deviation for Various Elements by Domain

Table 8-1: Univariate Statistics for Head Assays by Domain

Domain	Statistic	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	LOI	MgO	MnO	P_2O_5	REC
1	Num Rec	403	403	403	403	403	403	402	403	403	403	403
	Mean	10.47	11.21	54.54	1.07	10.44	1.17	3.07	5.01	0.18	0.21	93.48
	Std Devn	5.81	2.53	8.42	0.56	4.16	0.40	2.90	2.09	0.08	0.07	9.47
2	Num Rec	528	528	528	528	528	528	528	528	528	528	528
	Mean	7.42	13.53	49.81	0.77	13.82	1.27	5.59	4.22	0.16	0.23	86.39
	Std Devn	1.83	3.77	8.54	0.17	6.97	0.48	5.30	1.74	0.05	0.04	13.90
3	Num Rec	702	702	702	702	702	702	702	702	702	702	702
	Mean	11.39	12.97	51.57	1.16	10.68	1.21	2.38	5.15	0.20	0.26	97.90
	Std Devn	5.95	2.86	5.90	0.58	2.95	0.44	1.78	2.27	0.08	0.05	3.42
4	Num Rec	219	219	219	219	219	219	219	219	219	219	219
	Mean	9.30	11.46	45.48	0.89	17.11	0.89	6.12	5.97	0.20	0.25	87.48
	Std Devn	2.74	1.97	4.33	0.26	3.45	0.21	3.68	1.21	0.04	0.04	9.90
5	Num Rec	65	65	65	65	65	65	65	65	65	65	65
	Mean	7.33	14.40	52.88	0.78	11.39	1.41	3.95	4.05	0.15	0.24	89.82
	Std Devn	2.41	2.58	7.34	0.20	4.99	0.38	4.42	1.88	0.06	0.05	10.78
6	Num Rec	515	515	515	515	515	515	515	515	515	515	515
	Mean	8.99	12.96	54.97	0.93	10.18	1.26	2.93	4.35	0.17	0.21	95.25
	Std Devn	4.29	2.52	7.20	0.42	3.47	0.38	1.61	1.73	0.06	0.06	6.91
7	Num Rec	637	637	637	637	637	637	637	637	637	637	637
	Mean	8.17	13.47	51.11	0.83	12.93	1.25	4.05	4.76	0.17	0.23	88.80
	Std Devn	2.23	3.49	7.56	0.20	5.96	0.45	4.38	1.83	0.05	0.04	13.30
8	Num Rec	197	197	197	197	197	197	197	197	197	197	197
	Mean	7.10	9.63	57.40	0.65	11.29	1.03	5.10	5.16	0.14	0.13	91.39
	Std Devn	2.53	1.56	6.44	0.19	4.10	0.35	2.15	2.28	0.06	0.03	6.96

Domain	Statistic	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	LOI	MgO	MnO	P_2O_5	REC
9	Num Rec	262	262	262	262	262	262	262	262	262	262	262
	Mean	7.40	15.59	51.58	0.76	12.09	1.39	3.13	4.27	0.16	0.25	91.60
	Std Devn	2.11	2.92	7.67	0.19	5.71	0.35	5.78	1.67	0.05	0.03	7.54

Table 8-2: Univariate Statistics for Davis Tube Assays by Domain

Domain	Statistic	DTR	Fe	Al_2O_3	SiO_2	Ti	CaO	K_2O	MgO	Mn	P	LOI
1	Num Rec	277	267	267	267	267	267	267	267	267	267	273
	Mean	5.09	56.28	3.79	4.84	4.94	1.16	0.14	3.28	0.51	0.10	-2.73
	Std Devn	4.25	2.64	0.31	2.73	0.15	0.42	0.11	0.23	0.01	0.02	0.50
3	Num Rec	650	641	641	641	641	641	641	641	641	641	650
	Mean	7.48	55.55	3.80	5.40	5.06	1.20	0.18	3.25	0.51	0.12	-2.50
	Std Devn	6.31	2.43	0.35	2.63	0.14	0.35	0.12	0.21	0.01	0.02	0.57
6	Num Rec	298	287	287	287	287	287	287	287	287	287	297
	Mean	4.46	55.66	3.81	5.51	4.97	1.23	0.18	3.25	0.51	0.11	-2.66
	Std Devn	3.47	2.07	0.26	2.12	0.16	0.35	0.10	0.22	0.01	0.02	1.03
7	Num Rec	15	14	14	14	14	14	14	14	14	14	15
	Mean	3.41	57.99	3.71	3.14	4.72	1.02	0.06	3.34	0.50	0.07	-2.99
	Std Devn	2.05	0.93	0.11	0.95	0.20	0.27	0.03	0.26	0.02	0.01	0.54
9	Num Rec	30	30	30	30	30	30	30	30	30	30	30
	Mean	4.66	53.41	4.01	7.53	5.02	1.58	0.26	3.45	0.51	0.14	-2.13
	Std Devn	1.89	1.88	0.27	2.02	0.10	0.26	0.10	0.14	0.01	0.02	0.34

8.4 Population Distributions

The sample populations are relatively homogenous for most domains. Figure 8-4 illustrates population distributions by domain for accumulated Fe_2O_3, SiO_2, Al_2O_3 and TiO_2 head grades using cumulative log probability plots. The domains exhibit no outlier samples that will require management during the estimation by top cutting or spatial restraint. The higher grade domains show a break in the population around 8% Fe_2O_3. The change in the population is not significant enough to warrant further domaining in the current resource but should be investigated as a potential high grade domain in future work.

Figure 8-5 provides examples of probability plots of accumulated Davis Tube Concentrate analyses including DTR, Fe, SiO_2 and Al_2O_3. Both the Fe and Al_2O_3 show a tight range of values with minimal variability. The range of DTR and SiO_2 values are much wider and with greater variability in the final concentrate.

Appendix F includes the probability plots for all analytes and domains assessed.

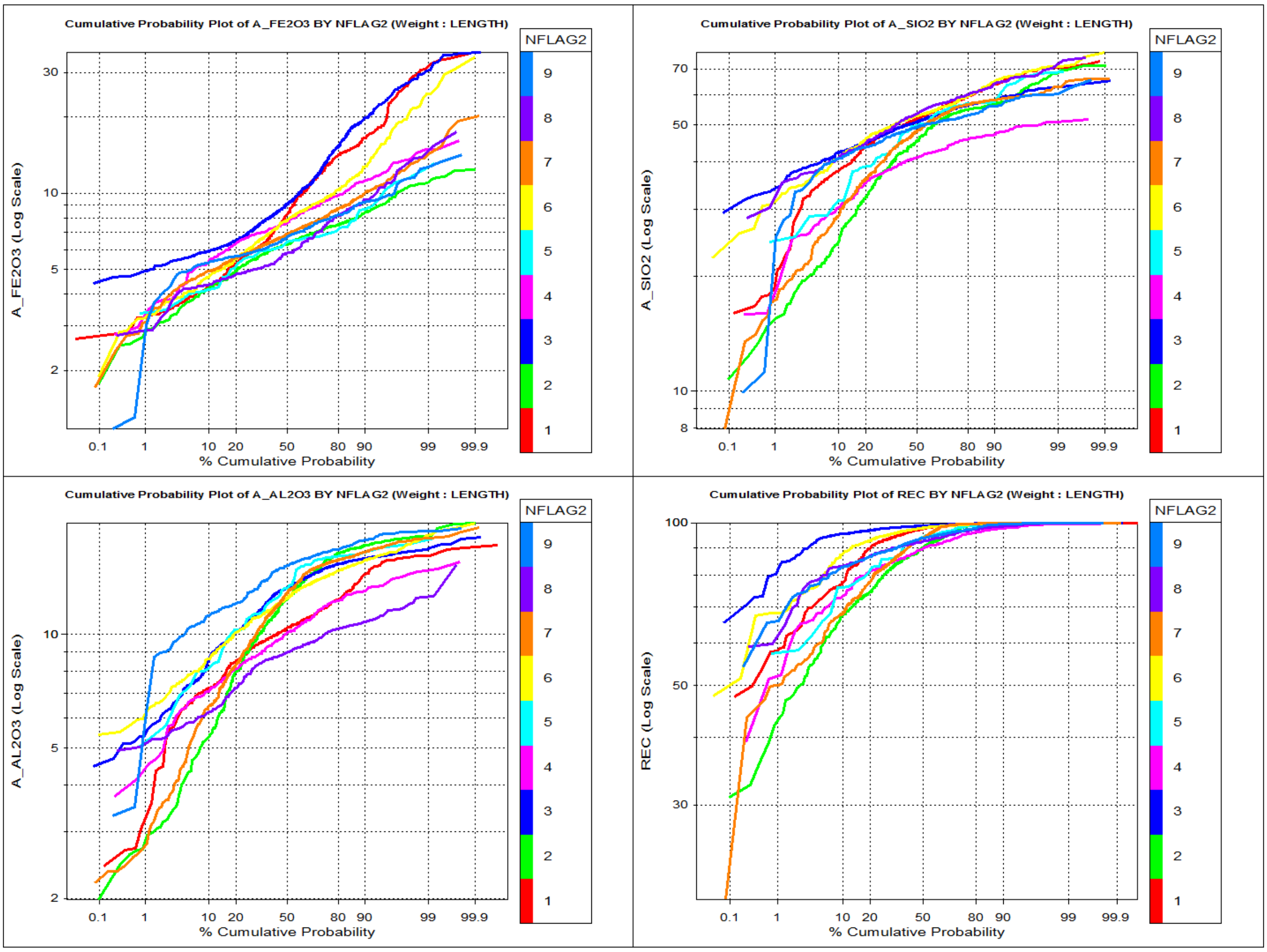


Figure 8-4: Cumulative Log Probability Plots of Selected Head Assays by Domain

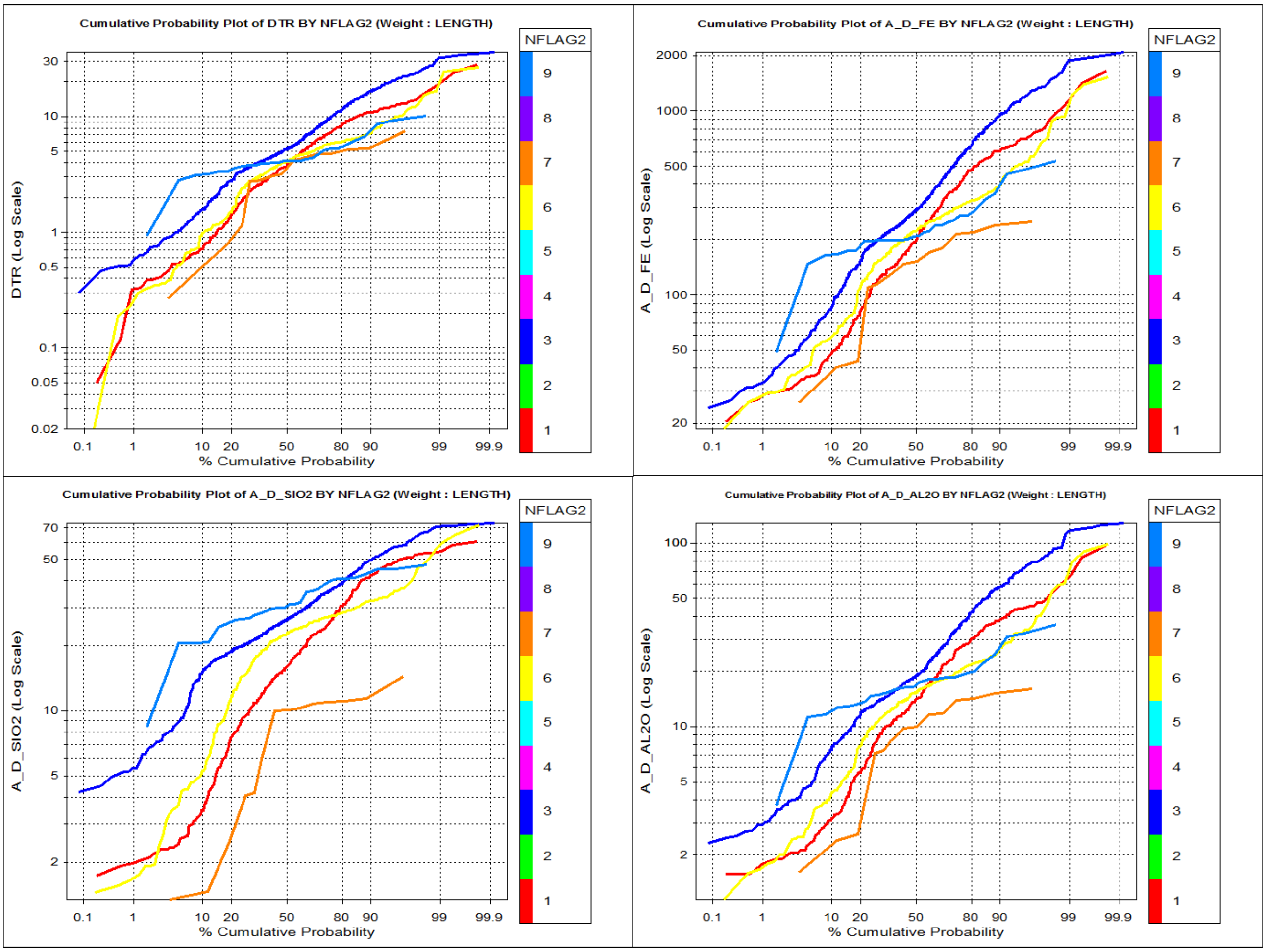


Figure 8-5: Cumulative Log Probability Plots of Selected Davis Tube Measurements by Domain

8.5 Analyte Correlation

Scatter plots were generated for all analytes. Generally the plots show clean data sets with the expected correlations. Correlation matrices were also created and studied. These were initially produced on the combined domain dataset (see Table 8-4 to Table 8-6). For the Head assays, the correlation coefficient was substantially improved for some elements when confined to individual domains. Table 8-3 details the relationships between the various Head assays as suggested by the analysis.

For the Head Assays in relation to Davis Tube assays the strongest correlation (i.e. Abs(R)>0.7) is observed between DTR and Fe_2O_3, DTR and TiO_2 and between DTR and MnO, For Davis Tube Concentrate analytes; however, the strongest correlation (i.e. Abs(R)>0.7) is between DT_Fe_2O_3 and DT_Al_2O_3, DT_Fe_2O_3 and DT_SiO_2, DT_Fe_2O_3 and DT_CaO, DT_Fe_2O_3 and K_2O.

Table 8-3: Head Assays that Show Strong to Medium Relationship Shared by Most Domains

Strong Relationship (where minimum of 8 domains have absolute value of R > 0.7)		**Moderate Relationship (where minimum of 5 domains have absolute value of R > 0.7)**	
Variable1	**Variable 2**	**Variable1**	**Variable 2**
Fe_2O_3	TiO_2	Al_2O_3	SiO_2
Fe_2O_3	MnO	LOI	CaO
Fe_2O_3	MgO	Al_2O_3	K_2O
SiO_2	K_2O	MgO	K_2O^-
TiO_2	MnO	Al_2O_3	CaO^-
MnO	MgO		
Fe_2O_3	DTR*		
TiO2	DTR*		
MnO	DTR*		
CaO	SiO2$^-$		
CaO	K2O$^-$		

*Only five domains with results

-Negative Correlation

A number of elements have near perfect correlation. In particular the Fe_2O_3 head assay presents a remarkable positive correlation with the DTR.

Figure 8-6 provides the scatter diagram between these two elements. In previous years a single linear correlation has been used. With the additional DTR data included this year a change in the slope of the correlation occurs around 20% Fe_2O_3. To accommodate this, two regression equations were used to estimate DTR where no data was available.

The correlation coefficient is 0.93 for both regressions and the equations are:

$$\text{Fe2O3} < 20\ \% : DTR = 0.9155 * Fe_2O_3 - 3.3494$$

$$\text{Fe2O3} > 20\ \% : DTR = 1.1944 * Fe_2O_3 - 8.3649$$

This suggests that the DTR for non-tested other sample intervals can be predicted with good confidence using the Fe_2O_3 head assays.

Table 8-4: Correlation Matrix for Selected Head Assays (Domains 1 to 9 Combined)

	Fe_2O_3	Al_2O_3	SiO_2	P_2O_5	LOI	TiO_2	CaO	MnO	MgO	K_2O	REC	DTR
Fe_2O_3	1.00	-0.47	-0.48	0.55	-0.28	0.99	0.13	0.89	0.69	-0.54	0.07	0.98
Al_2O_3	-0.47	1.00	0.50	0.04	-0.43	-0.39	-0.59	-0.49	-0.66	0.83	0.24	-0.64
SiO_2	-0.48	0.50	1.00	-0.56	-0.56	-0.40	-0.88	-0.59	-0.61	0.76	0.43	-0.74
P_2O_5	0.55	0.04	-0.56	1.00	-0.17	0.53	0.25	0.62	0.50	-0.27	-0.07	0.63
LOI	-0.28	-0.43	-0.56	-0.17	1.00	-0.33	0.74	-0.24	-0.05	-0.38	-0.44	-0.46
TiO_2	0.99	-0.39	-0.40	0.53	-0.33	1.00	0.02	0.84	0.59	-0.43	0.13	0.99
CaO	0.13	-0.59	-0.88	0.25	0.74	0.02	1.00	0.34	0.53	-0.80	-0.55	0.34
MnO	0.89	-0.49	-0.59	0.62	-0.24	0.84	0.34	1.00	0.89	-0.69	-0.07	0.87
MgO	0.69	-0.66	-0.61	0.50	-0.05	0.59	0.53	0.89	1.00	-0.82	-0.21	0.66
K_2O	-0.54	0.83	0.76	-0.27	-0.38	-0.43	-0.80	-0.69	-0.82	1.00	0.34	-0.66
REC	0.07	0.24	0.43	-0.07	-0.44	0.13	-0.55	-0.07	-0.21	0.34	1.00	0.07
DTR	0.98	-0.64	-0.74	0.63	-0.46	0.99	0.34	0.87	0.66	-0.66	0.07	1.00
R>=	**-1.00**	**-0.80**	**-0.60**	**-0.40**	**-0.20**	**0.00**	**0.20**	**0.40**	**0.60**	**0.80**	**1.00**	

Table 8-5: Correlation Matrix for Selected Head Assays vs. DT Measurements (Domains 1 to 9 Combined)

	DTR Fe	DTR Al_2O_3	DTR SiO_2	DTR P	DTR LOI	DTR Ti	DTR CaO	DTR MgO	DTR Mn	DTR K_2O
Fe_2O_3	0.47	-0.38	-0.53	-0.26	-0.21	0.49	-0.42	0.05	0.12	-0.51
Al_2O_3	-0.65	0.52	0.68	0.67	0.09	-0.13	0.46	-0.17	-0.02	0.74
SiO_2	-0.22	0.19	0.31	0.13	-0.15	-0.47	0.10	-0.28	0.00	0.35
P_2O_5	-0.05	0.02	-0.04	0.16	-0.08	0.43	0.13	0.35	-0.07	-0.09
LOI	-0.17	0.25	0.17	0.02	0.24	-0.23	0.13	-0.18	-0.11	0.19
TiO_2	0.46	-0.36	-0.51	-0.22	-0.23	0.49	-0.44	-0.01	0.14	-0.47
CaO	0.22	-0.25	-0.29	-0.33	0.21	0.23	0.05	0.54	-0.07	-0.44
MgO	0.35	-0.35	-0.42	-0.38	-0.05	0.30	-0.11	0.51	-0.07	-0.54
MnO	0.40	-0.38	-0.47	-0.27	-0.11	0.46	-0.24	0.32	0.05	-0.53
K_2O	-0.55	0.54	0.61	0.52	0.01	-0.38	0.29	-0.38	-0.10	0.72
REC	-0.13	0.00	0.14	0.40	-0.29	0.21	0.04	-0.15	0.21	0.19
DTR	0.49	-0.39	-0.54	-0.24	-0.10	0.49	-0.46	-0.01	0.17	-0.50
R>=	**-1.00**	**-0.80**	**-0.60**	**-0.40**	**-0.20**	**0.00**	**0.20**	**0.40**	**0.60**	**0.80**

Table 8-6: Correlation Matrix for Selected DT Measurements (Domains 1 to 9 Combined)

	Fe	Al_2O_3	SiO_2	P	LOI	Ti	CaO	MgO	Mn	Na	K_2O
Fe	1.00	-0.88	-0.99	-0.73	-0.76	0.47	-0.91	-0.33	0.51	-0.92	-0.94
Al_2O_3	-0.88	1.00	0.88	0.55	0.68	-0.60	0.73	0.20	-0.63	0.83	0.88
SiO_2	-0.99	0.88	1.00	0.70	0.69	-0.53	0.89	0.27	-0.50	0.94	0.96
P	-0.73	0.55	0.70	1.00	0.67	0.01	0.59	0.02	-0.07	0.68	0.75
LOI	-0.76	0.68	0.69	0.67	1.00	-0.10	0.64	0.19	-0.38	0.66	0.70
Ti	0.47	-0.60	-0.53	0.01	-0.10	1.00	-0.48	-0.22	0.63	-0.48	-0.45
CaO	-0.91	0.73	0.89	0.59	0.64	-0.48	1.00	0.64	-0.53	0.76	0.73
MgO	-0.33	0.20	0.27	0.02	0.19	-0.22	0.64	1.00	-0.45	0.09	0.03
Mn	0.51	-0.63	-0.50	-0.07	-0.38	0.63	-0.53	-0.45	1.00	-0.45	-0.42
Na	-0.92	0.83	0.94	0.68	0.66	-0.48	0.76	0.09	-0.45	1.00	0.93
K_2O	-0.94	0.88	0.96	0.75	0.70	-0.45	0.73	0.03	-0.42	0.93	1.00
R>=	**-1.00**	**-0.80**	**-0.60**	**-0.40**	**-0.20**	**0.00**	**0.20**	**0.40**	**0.60**	**0.80**	**1.00**

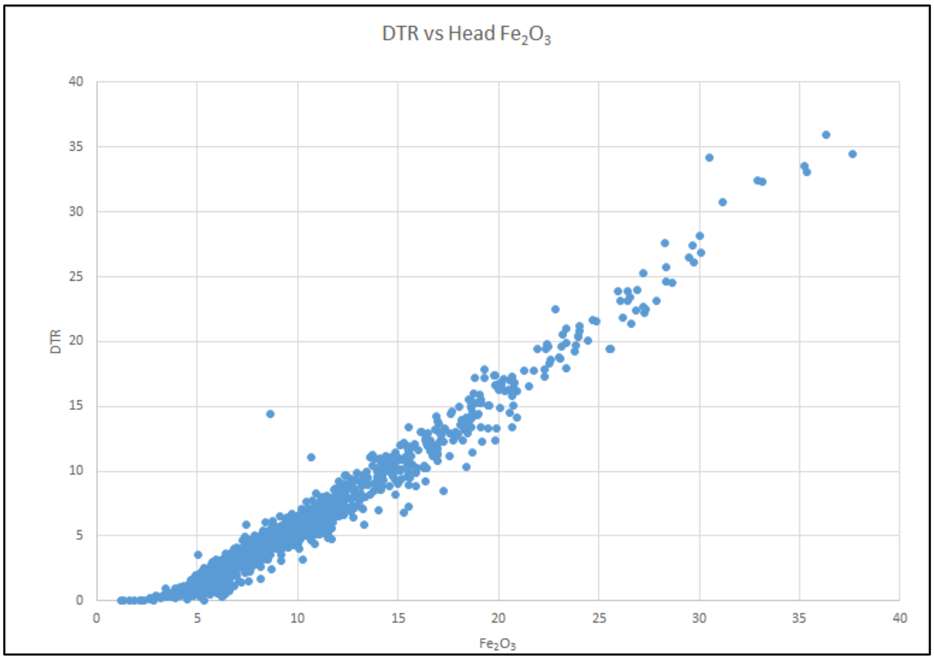


Figure 8-6: Scatter plot for Fe_2O_3 vs. DTR

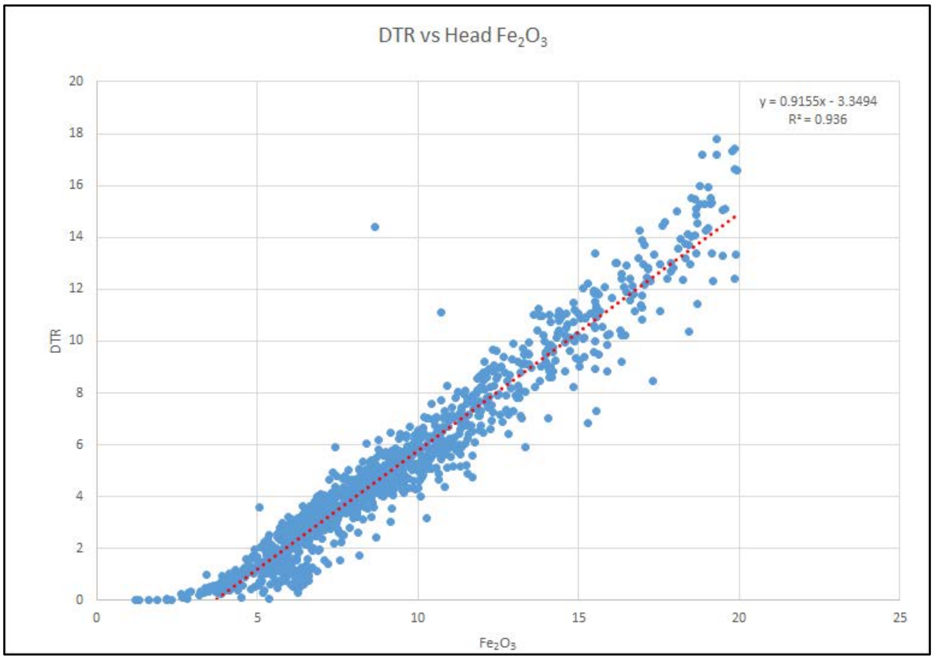


Figure 8-7: Scatter plot for Fe_2O_3 vs. DTR Fe_2O_3 less than 20%

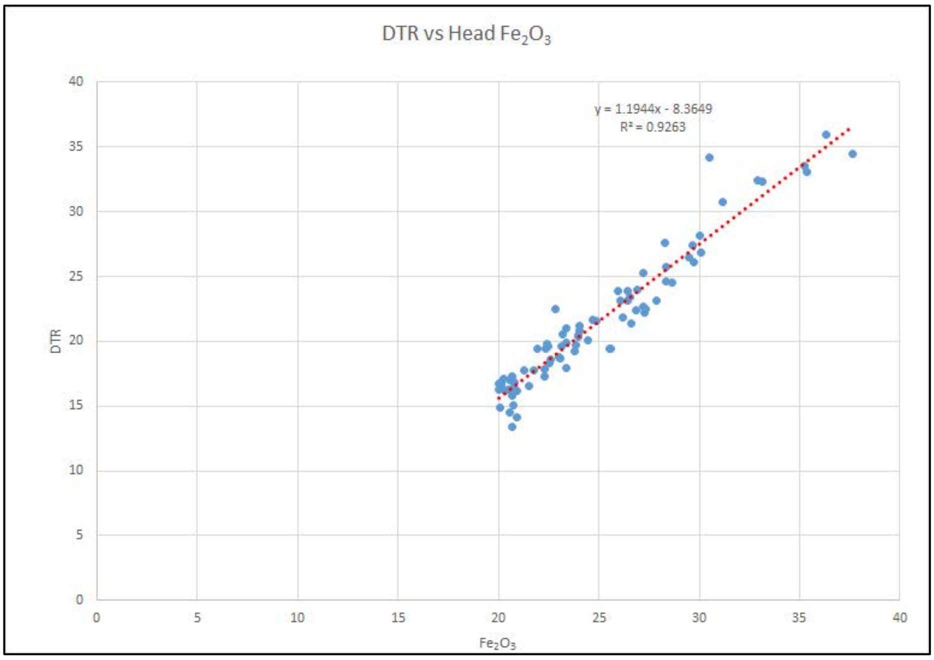


Figure 8-8: Scatter plot for Fe_2O_3 vs. DTR Fe_2O_3 greater than 20%

Appendix F includes the scatter plots for all analytes and domains assessed.

9.0 VARIOGRAPHIC ANALYSIS

9.1 Variography Objectives and Approach

Variography was undertaken for the previous studies to model spatial continuity of the variables for the South Taranaki project. Variography was previously carried out for the head variables: Fe_2O_3, SiO_2, Al_2O_3, LOI, P_2O_5, TiO_2, REC and V.

In 2013, additional holes were drilled primarily in Domain 1, Domain 2, Domain 3, Domain 4, Domain 6, and Domain 7 (Figure 9-1). The amount of additional data is only likely to have an impact on the variography for Domains 2 and 3.

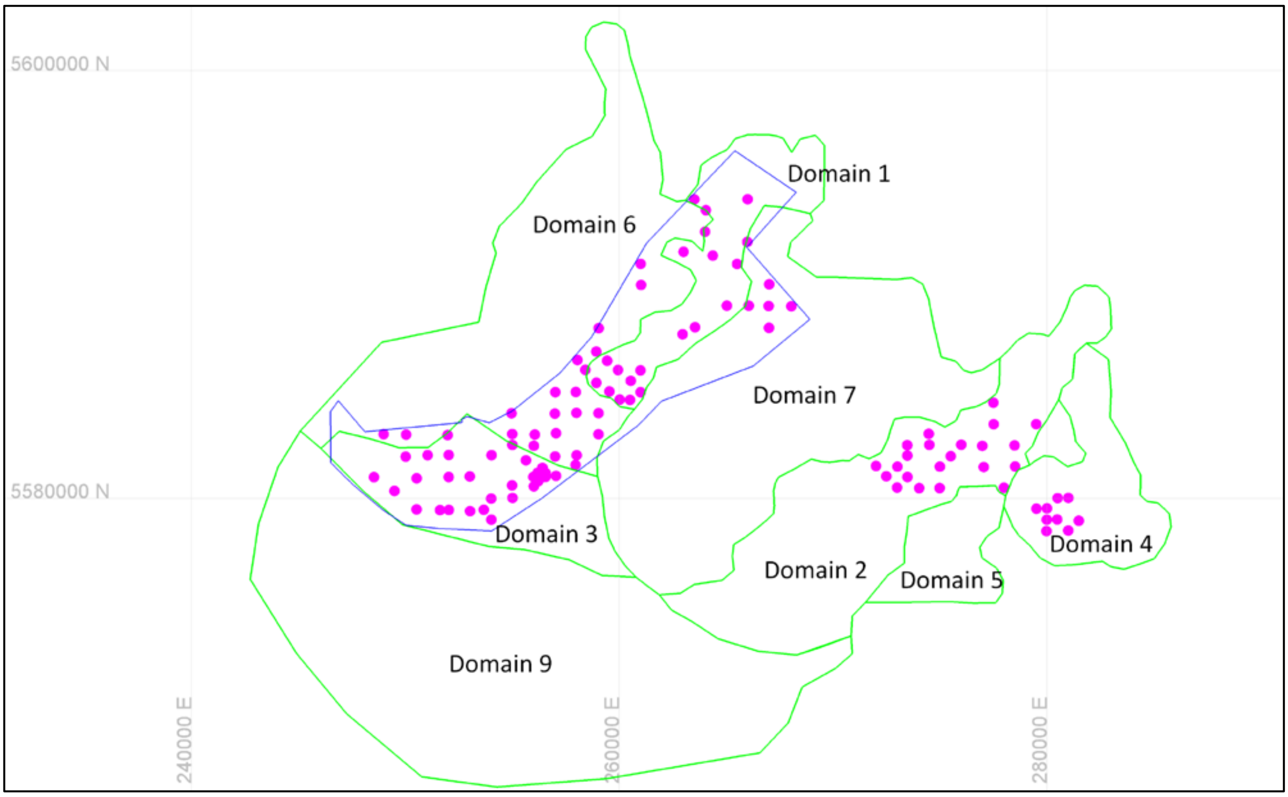


Figure 9-1: 2013 Drilling

Similarly the DTC samples are concentrated within the initial mining area, covering some of Domain 1, Domain 3, Domain 6 and Domain 7. Due to insufficient samples for parts of these domains, variography for the DTC samples was carried out using a mining area combined composite dataset.

As part of the sample preparation for analysis the head samples were first screened removing all material greater than 2 mm. As such the assays are only representative of the recovered portion (i.e. Rec%) that is recorded in the database.

Similarly, for the DTC assays, only the screened sample was processed through a Davis Tube magnetic recovery system before being analysed for various concentrate assays. Therefore, the DTC assays are only representative of the recovered magnetite portion of the screened material and must be weighted by Rec*DTR.

For variography and estimation purposes therefore, the composite concentrate grades were all multiplied by their respective Rec*DTR to produce accumulated concentrate grades. These accumulated concentrate grades were then used for variographic analysis and for grade estimation. Weighting the concentrate grades by their Rec*DTR value, results in a more realistic estimate of the concentrate grades in the final model. The accumulated concentrate grades are named ACC_DT_<DTC analyte>.

An accumulation was also used for the head assays. The Head assays were multiplied by the Rec%. The accumulated head grades are named ACC_<Head analyte>.

In the current study the variography was performed in Golder proprietary software, which allows sills and ranges to be modelled to reflect the zonal and geometric anisotropy in the variograms.

The objectives of the variography were to:

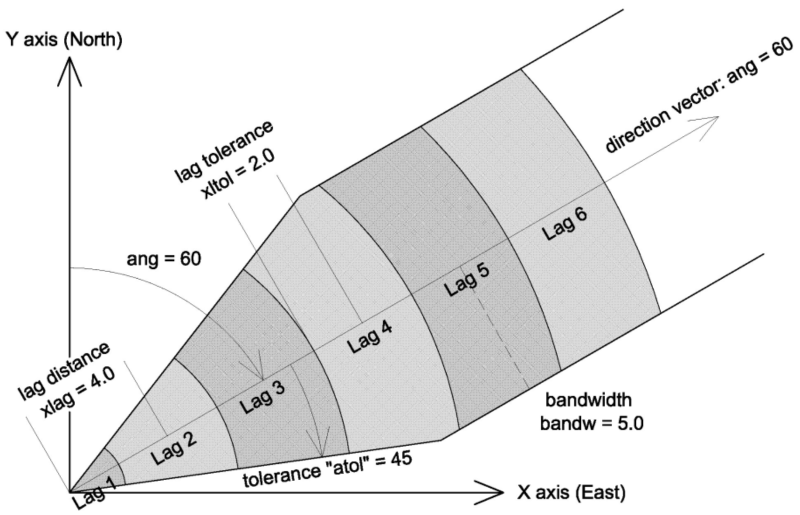
- Establish the directions of major grade continuity for each element in the domains
- Quantify spatial continuity (variability, anisotropy and overall continuity), and
- Provide variogram model parameters for use in geostatistical grade interpolation (Ordinary Kriging).

An overview of the variography procedure for the data set used is as follows:

- The angular and distance search tolerances used for various domains are provided in Table 9-1. Depending on the average data spacing and data arrangements, the angular and distance search tolerances were varied for different domains.
- Directional variograms for all domains were calculated using a 500 m lag interval. The drill hole spacing is variable with the densest areas having an approximate 500 m × 500 m spacing. The average spacing is greater than 1000 m.
- The nugget variances for all domains were modelled from average downhole variograms based on a 1 m lag, reflecting the downhole composite spacing. The downhole variogram model provided the nugget and under normal circumstances would approximate the minor axis for the 3D variogram model.
- In most cases, correlograms were used for variogram modelling, as they showed clearer interpretable structures. For Domains 5 insufficient data was available for modelling.
- Variography was carried out for the Head assays as ACC_Fe2O3, ACC_Al2O3, ACC_SiO2, ACC_P2O5, ACC_LOI, ACC_Mgo, ACC_K2O, ACC_MnO, ACC_TiO2, ACC_CaO and Rec%. For the Davis Tube concentrate the elements considered are; ACC_DD_Fe2O3, ACC_ DD_Al2O3, ACC_ DD_SiO2, ACC_ DD_P2O5, ACC_ DD_LOI, ACC_ DD_Mgo, ACC_ DD_K2O, ACC_ DD_MnO, ACC_ DD_TiO2, ACC_ DD_CaO and ACC_DTR (i.e. Rec*DTR).

Table 9-1: Parameters Used for Variographic Analysis

Parameter	1	2	3	4	5	6	7	8	9	DD
Horizontal Angle Tolerance (atol)	50°	50°	50°	50°	50°	20°	20°	20°	30°	20°
Vertical Angle Tolerance (vtol)	10°	10°	5°	5°	5°	5°	5°	5°	5°	5°
Horizontal Distance Bandwidth (bandw) – Metres	1000	1000	1000	1000	1000	2000	2000	2000	2000	2000
Vertical Distance Bandwidth – Metres	10	10	3	3	3	3	3	3	2	3
Lag Distance (xlag) – Metres	500	500	500	500	500	500	500	500	500	500
Lag Tolerance (xltol) – Metres	250	250	250	250	250	250	250	250	250	250



NB: The tolerance values on Figure 9-2 are generic. Actual values used are provided in Table 9-1.

Figure 9-2: Definition of Variogram Parameters in Table 9-1

9.2 Variography Results

Across all analytes the nugget is very low and ranges are in excess of 1000 m. Figure 9-3 and Figure 9-4 illustrate typical variograms from Domain 3.

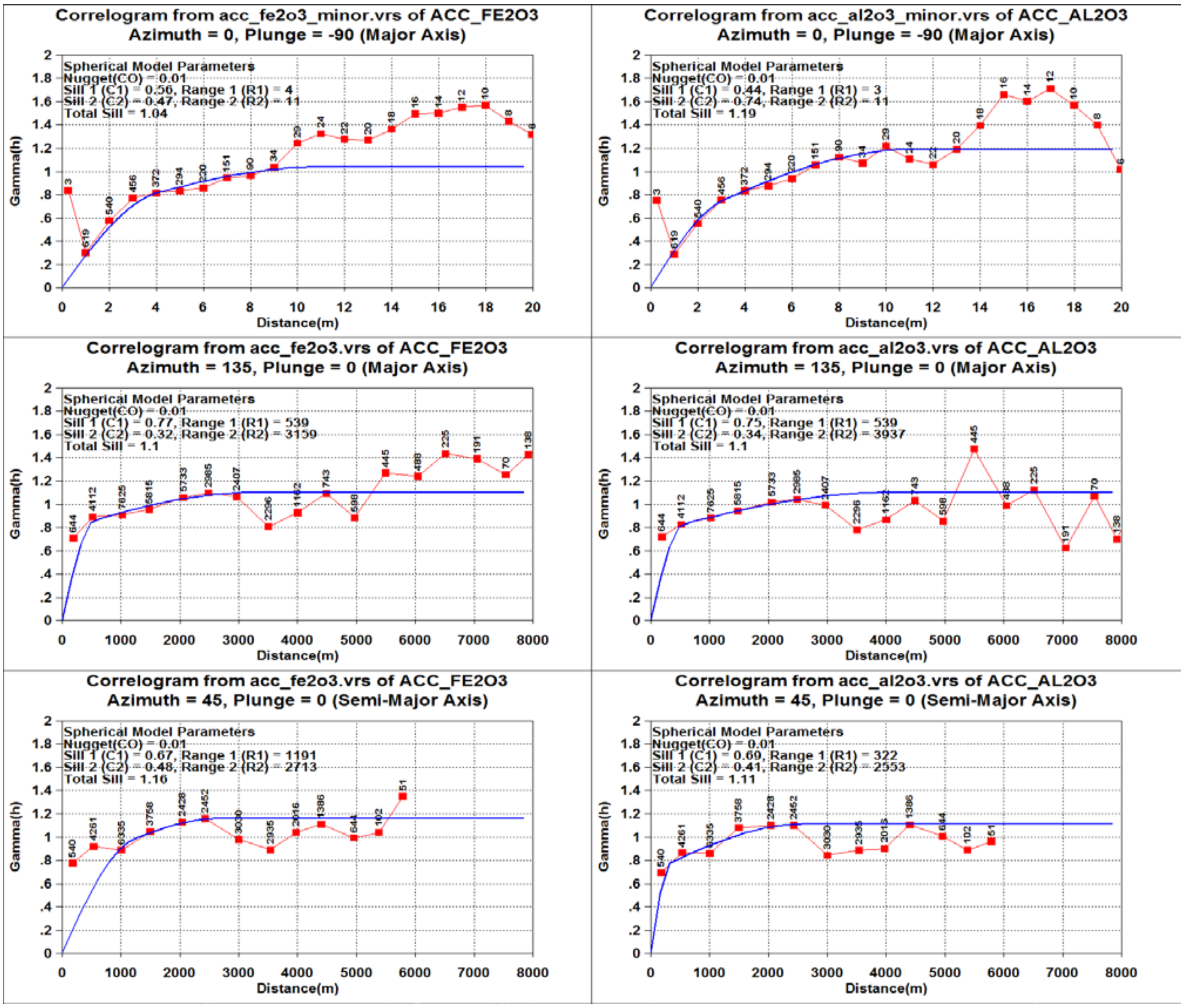


Figure 9-3: Variograms – Domain 3 Fe_2O_3 and Al_2O_3

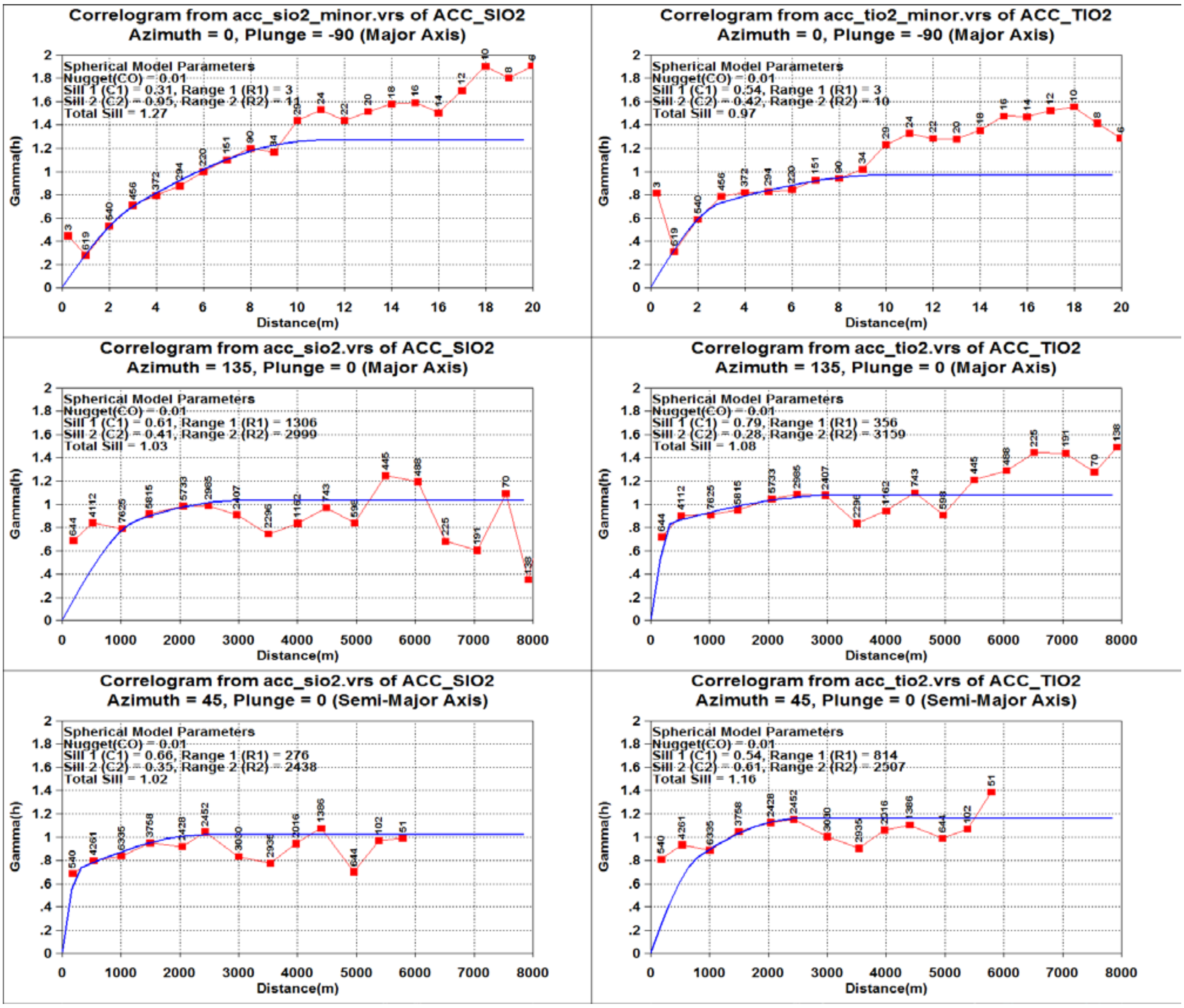


Figure 9-4: Variograms – Domain 3 SiO_2 and TiO_2

The results of the variographic analysis were compiled and the nugget, sill and range values applied as kriging parameters during the grade interpolation.

The complete set of plots for the variograms assessed is included in Appendix G.

10.0 GRADE INTERPOLATION

10.1 Grade Interpolation Plan

Grade estimation for the South Taranaki deposit was carried out using linear estimation methods. A three pass estimation plan was used with a single sector (no octant search) for all estimation groups using Ordinary Kriging for Area 2 and Koitiata. For head grades, ACC_Fe2O3, ACC_Al2O3, ACC_SiO2, ACC_P2O5, ACC_LOI, ACC_Mgo, ACC_K2O, ACC_MnO, ACC_TiO2, ACC_CaO and Rec% were estimated. For the DTC the elements estimated were; ACC_DD_Fe2O3, ACC_ DD_Al2O3, ACC_ DD_SiO2, ACC_ DD_P2O5, ACC_ DD_LOI, ACC_ DD_Mgo, ACC_ DD_K2O, ACC_ DD_MnO, ACC_ DD_TiO2, ACC_ DD_CaO, ACC_DTR (i.e. Rec*DTR). An additional DTR variable, DTR_EST, was estimated which used all available DTR analysis data and values calculated from the $Fe2O_3$ vs DTR regression formula where analysis data was absent. The waste domains were not estimated.

In order to maintain acceptable total assay values, the same search neighbourhood orientations and dimensions were used for each variable in each domain.

Estimation panel sizes were governed by the parent block size, which was 300 m by 300 m by 1 m.

No high grade restraining or sample cutting was applied during the estimation. Analysis of the data set indicated no significant outlier samples that would have an undue influence in the estimation.

Soft boundaries were used between domains so that a domain was estimated using all data from adjacent domains.

Search ellipses were oriented based on the geological interpretation of the domain and the three pass estimation incrementally increase search distances to ensure adequate samples were found to make an estimate for all blocks.

The estimation plan parameters used for grade interpolation of all variables for Area 2 and Koitiata are summarised in Table 10-1.

Table 10-1: Estimation Parameters

Estimation Method	Ordinary Kriging
Variables Estimated	ACC_Fe2O3, ACC_Al2O3, ACC_SiO2, ACC_P2O5, ACC_LOI, ACC_Mgo, ACC_K2O, ACC_MnO, ACC_TiO2, ACC_CaO and Rec%. For the Davis Tube concentrate the elements considered are; ACC_DD_Fe2O3, ACC_ DD_Al2O3, ACC_ DD_SiO2, ACC_ DD_P2O5, ACC_ DD_LOI, ACC_ DD_Mgo, ACC_ DD_K2O, ACC_ DD_MnO, ACC_ DD_TiO2, ACC_ DD_CaO, ACC_DTR (i.e. Rec*DTR), DTR_EST.
Search Radius Pass 1 (x/y/z)	1000 m × 500 m × 2 m 2000 m × 1000 m × 4 m 4000 m × 4000 m × 20 m
Anisotropy	Set by krige parameters
Estimation Panel Size (X/Y/Z)	300 m × 300 m × 1 m
Discretisation (X points/Y points/Z points)	5/5/1
Search Volume Geometry	Ellipsoid
Search type	Normal
Minimum No. of Samples	1
Maximum No. of Samples	32
Maximum No. of Samples per Hole	2
Unfolding surface	bathymetric surface (bathy_topo_sp.00t)

10.2 Density Assignment

The *in situ* bulk density was calculated using the Fe regression developed from the calculated theoretical bulk density corrected for the measured results. After estimation of the Fe_2O_3 grades the dry bulk density was calculated by the formula ((Fe_2O_3*0.6994)+81.191)/51.064 where Fe_2O_3 is 69.94% Fe.

10.3 Validation of Grade Estimates

Statistical and visual assessment of the block model was undertaken to assess successful application of the various estimation passes and to ensure that as far as the data allowed, all blocks within mineralised domains were estimated and the model estimates considered acceptable.

In addition, a detailed review of the differences between the 2012 and 2013 models was undertaken. This review is included as Appendix I.

10.3.1 Visual Assessment of Grade Estimates

An onscreen validation between samples and blocks was completed on the model. The onscreen validation process involved comparing block estimates and composites grades in cross-section and in plan.
Figure 10-1 shows a cross-section of the block model and drill holes at 558400 mN section. The block model shows a good correlation with the drilling data. Note that some drill holes are a considerable distance off the section plane of the block model and may appear incorrectly positioned.

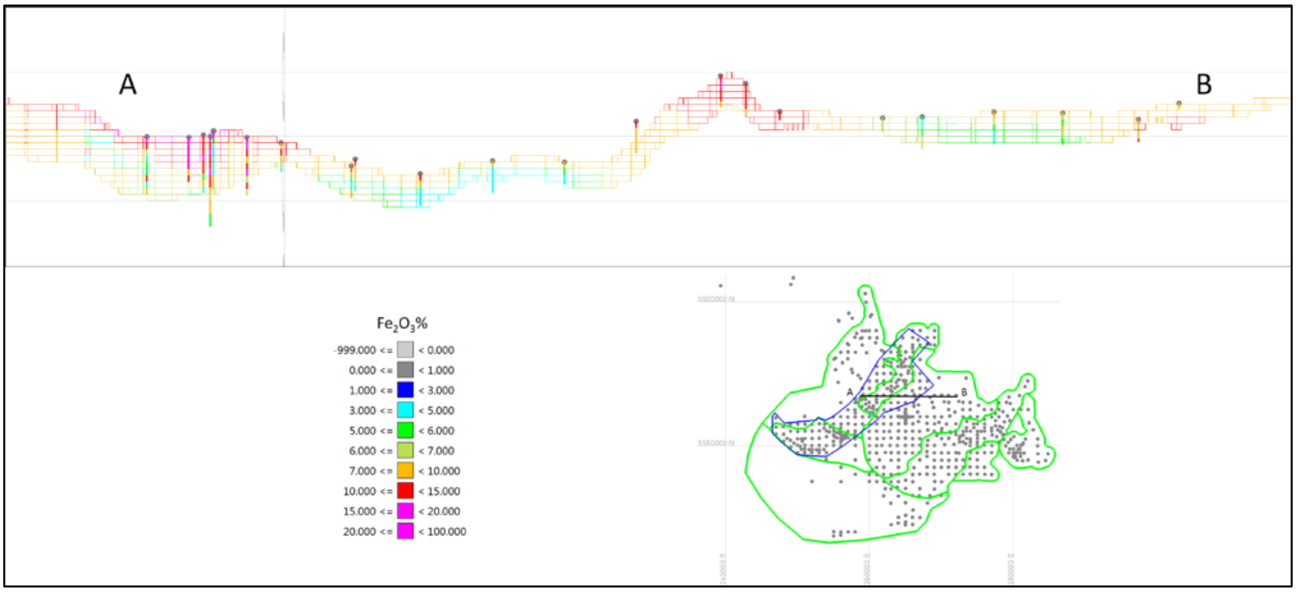


Figure 10-1: Sample Cross-Section at 5 586 870 mN (~100:1 Vertical Exaggeration)

10.3.2 Statistical Assessment of Grade Estimates

Statistical assessment of the grade estimates included mean grade reproduction checks. Table 10-2 compares the block model domain average to the length weighted composite average. The mean grade reproduction is considered acceptable. Concentrate grades were compared globally due to the incomplete sample set per domain.

Table 10-2: Head Grade – Declustered Accumulated Composites vs. Accumulated Model Grades

	Field	Domain							
		1	2	3	4	5	6	7	9
Composites	ACC_AL_2O_3	10.04	12.21	13.04	9.88	12.82	12.25	11.65	14.39
	ACC_CAO	9.29	11.07	10.29	15.20	10.07	9.62	11.43	10.73
	ACC_FE_2O_3	8.73	6.26	10.13	7.36	6.27	8.43	7.01	6.74
	ACC_K_2O	1.08	1.17	1.22	0.78	1.29	1.19	1.08	1.29
	ACC_LOI	0.10	4.38	2.37	5.94	3.87	2.73	3.64	2.58
	ACC_MGO	4.38	3.42	4.83	4.89	3.37	4.15	4.16	3.86
	ACC_MNO	0.16	0.13	0.19	0.16	0.13	0.16	0.15	0.14
	ACC_P_2O_5	0.17	0.20	0.25	0.21	0.21	0.20	0.20	0.23
	ACC_SIO_2	51.25	44.10	51.28	38.81	47.89	52.56	44.30	47.53
	ACC_TIO_2	0.95	0.66	1.07	0.71	0.67	0.88	0.71	0.70
	REC	91.51	86.50	97.62	85.99	89.69	95.04	86.95	91.33
Model	ACC_AL_2O_3	9.90	12.58	13.32	9.59	12.83	12.62	11.68	14.50
	ACC_CAO	9.48	11.14	10.21	15.33	10.14	9.57	11.51	10.15
	ACC_FE_2O_3	9.83	6.42	9.90	6.97	6.21	8.29	7.06	6.90
	ACC_K_2O	1.06	1.17	1.24	0.76	1.28	1.23	1.08	1.34
	ACC_LOI	3.17	4.15	2.26	6.36	3.84	2.81	3.72	2.41
	ACC_MGO	4.53	3.45	4.67	4.72	3.38	4.01	4.15	3.68
	ACC_MNO	0.17	0.13	0.18	0.16	0.13	0.15	0.15	0.14
	ACC_P_2O_5	0.18	0.20	0.24	0.20	0.21	0.20	0.19	0.23
	ACC_SIO_2	50.41	43.76	51.18	37.77	47.39	53.53	44.54	49.12
	ACC_TIO_2	1.00	0.67	1.02	0.67	0.66	0.88	0.71	0.72
	REC	92.36	86.67	97.34	84.38	89.39	96.20	87.78	92.59
Variance	ACC_AL_2O_3	101%	97%	98%	103%	100%	97%	100%	99%
	ACC_CAO	98%	99%	101%	99%	99%	101%	99%	106%
	ACC_FE_2O_3	89%	98%	102%	106%	101%	102%	99%	98%
	ACC_K_2O	102%	100%	98%	102%	101%	97%	100%	96%
	ACC_LOI	3%	106%	105%	93%	101%	97%	98%	107%
	ACC_MGO	97%	99%	103%	104%	100%	104%	100%	105%
	ACC_MNO	95%	97%	103%	104%	100%	102%	100%	101%
	ACC_P_2O_5	95%	98%	101%	106%	100%	99%	102%	101%
	ACC_SIO_2	102%	101%	100%	103%	101%	98%	99%	97%
	ACC_TIO_2	94%	98%	105%	106%	101%	101%	100%	96%
	REC	99%	100%	100%	102%	100%	99%	99%	99%

Table 10-3: Concentrate Grades – Accumulated Composites vs. Accumulated Model Grades

	Composites	Model	Variance
ACC_D_AL2O	19.08	20.14	106%
ACC_D_CAO	5.58	5.73	103%
ACC_D_FE	291.37	309.97	106%
ACC_D_K2O	0.66	0.65	99%
ACC_D_LOI	-15.74	-16.83	107%
ACC_D_MGO	16.78	17.68	105%
ACC_D_MN	2.62	2.78	106%
ACC_D_P	0.54	0.56	103%
ACC_D_SIO2	22.43	22.53	100%
ACC_D_TI	26.04	27.49	106%
ACC_DTR	5.15	5.62	109%

Swath validations were also used to validate the estimation. The average grade of blocks and composite drill samples from panels 1000 m by 1000 m by 3 m across the deposit in X, Y, Z dimensions and combined were compared graphically. Figure 10-2 illustrates the results for ACC_Fe_2O_3 in all domains of Area 2. A good correlation exits between the model and the composite samples. Some swath plots show significant differences in the panels at the edges of the plots. These differences represent the edges of the deposit and are usually the result of limited drill sampling coverage across the panel.

The swath validations show a good correlation between the model and composite samples indicating the estimation is a valid representation of the raw data.

The full suite of swath plots for the models is included in Appendix H.

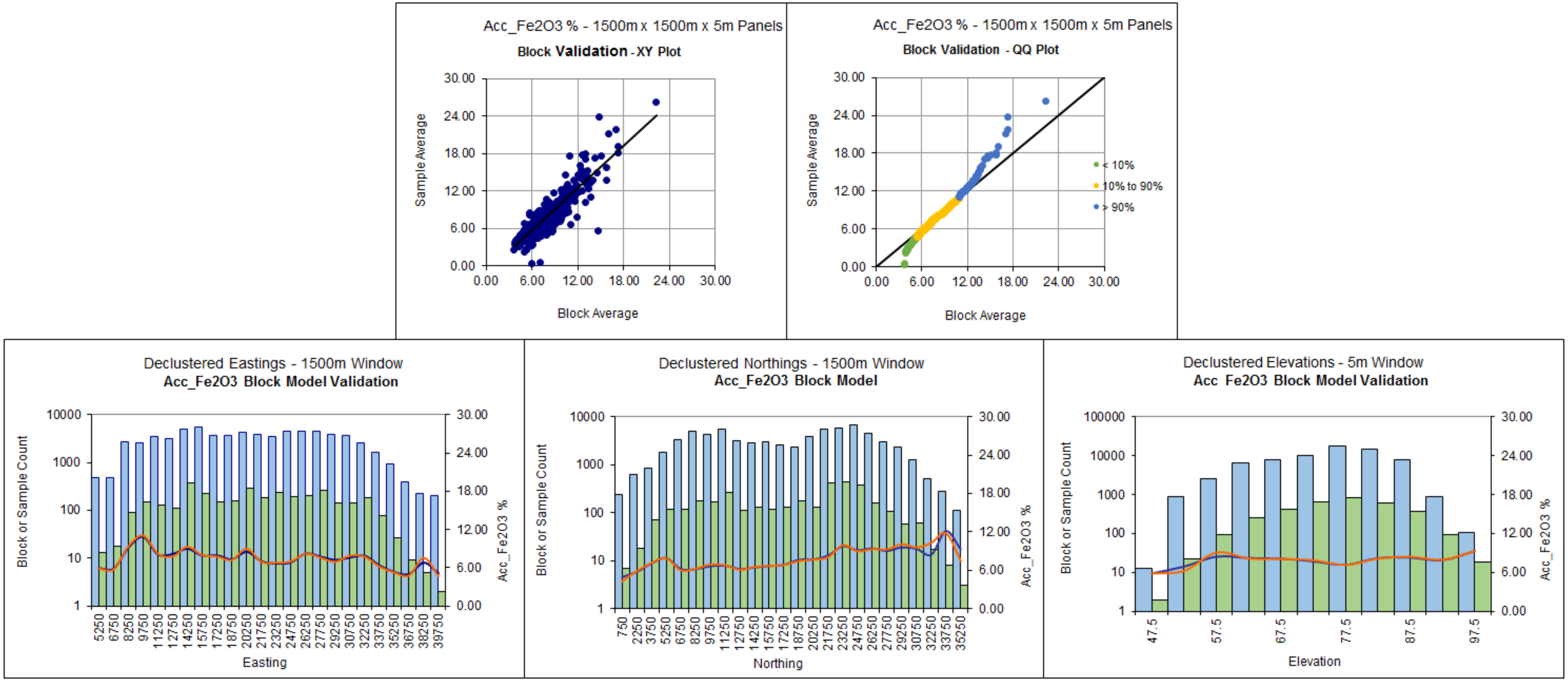


Figure 10-2: Swath Validation – Area 2 – ACC_Fe_2O_3

10.4 Mineral Resource Statement

The South Taranaki resource estimates are reported in accordance with the Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2012). The South Taranaki resource estimates have been prepared by employees of Golder Associates Pty Ltd. Golder and its employees are independent of Trans-Tasman Resources Ltd.

The South Taranaki estimate has been overseen by Stephen Godfrey, Principal Resource Geologist and Director of Resource Evaluation Services (RES). Mr Godfrey was subcontracted to Golder for this task. Mr Godfrey is a Member of the Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists. Mr Godfrey has undertaken the geological data analysis and constructed the geological model of the deposit. The estimation of the South Taranaki resource was undertaken by Dr Sia Khosrowshahi of Golder Associates. Dr Khosrowshahi is a geostatistician and Principal with Golder Associates and is a Member of the Australasian Institute of Mining and Metallurgy and a Member of the Australian Institute of Geoscientists.

Mr Godfrey has sufficient experience in alluvial deposits and depositional modelling to be considered a “Competent Person” as defined in the JORC Code (2012). Mr Godfrey has 30 years’ experience in the mining industry which includes exploration, modelling and mining activities variously for alluvial tin and gold deposits in Australia, ironsand deposits offshore in the Philippines and Australian coal deposits, all of which are applicable in contributing to the understanding of a deposit such as South Taranaki.

Dr Khosrowshahi has over 30 years’ experience in geology, ore reserves development, grade control, computerised mine planning systems and technical engineering activities related to projects in Australia, Chile, Indonesia and many other countries. He has specialist expertise in applying geostatistical methods to mineral resources, ore reserves and grade control. Dr Khosrowshahi has sufficient estimation experience to be considered a “Competent Person” as defined the JORC Code (2012).

The resource estimates were classified in accordance with the Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2012) as Indicated and Inferred based on drill holes available as of 4 September 2013.

The Mineral Resource is reported from the block *sia_dtr_est_post_b.bmf (Area 2 – domains 1-7 and 9)* and *south_acc_24_11_2012.bmf (Koitiata – domain 8)*. The Koitiata model was not updated in 2013.

The physical screened recovery has been applied to the models. Head grades and tonnages are for all material less than 2 mm in diameter. Concentrate grades are for the magnetically recoverable portion of the sample. Concentrate tonnage is calculated from the head tonnage and DTR.

The models have been reported at a 3.5% DTR cut-off grade where DTR analyses are available within the proposed mining area (Table 10-4 and Table 10-5). Outside this area a cut-off grade of 7.5% Fe_2O_3 has been used based on the statistical relationship between Fe_2O_3 and DTR (Table 10-6).

The resource model estimates have been classified as Indicated Resource where the drill spacing is on a 1000 m by 1000 m grid or closer, and Inferred Resource where the deposit is less systematically drilled but geological continuity can be interpreted. Appendix E contains the Golder drill spacing study from 2012 which statistically justifies the drill spacing limits applied. Figure 10-3 illustrates the regions applied to classify the 2013 resource.

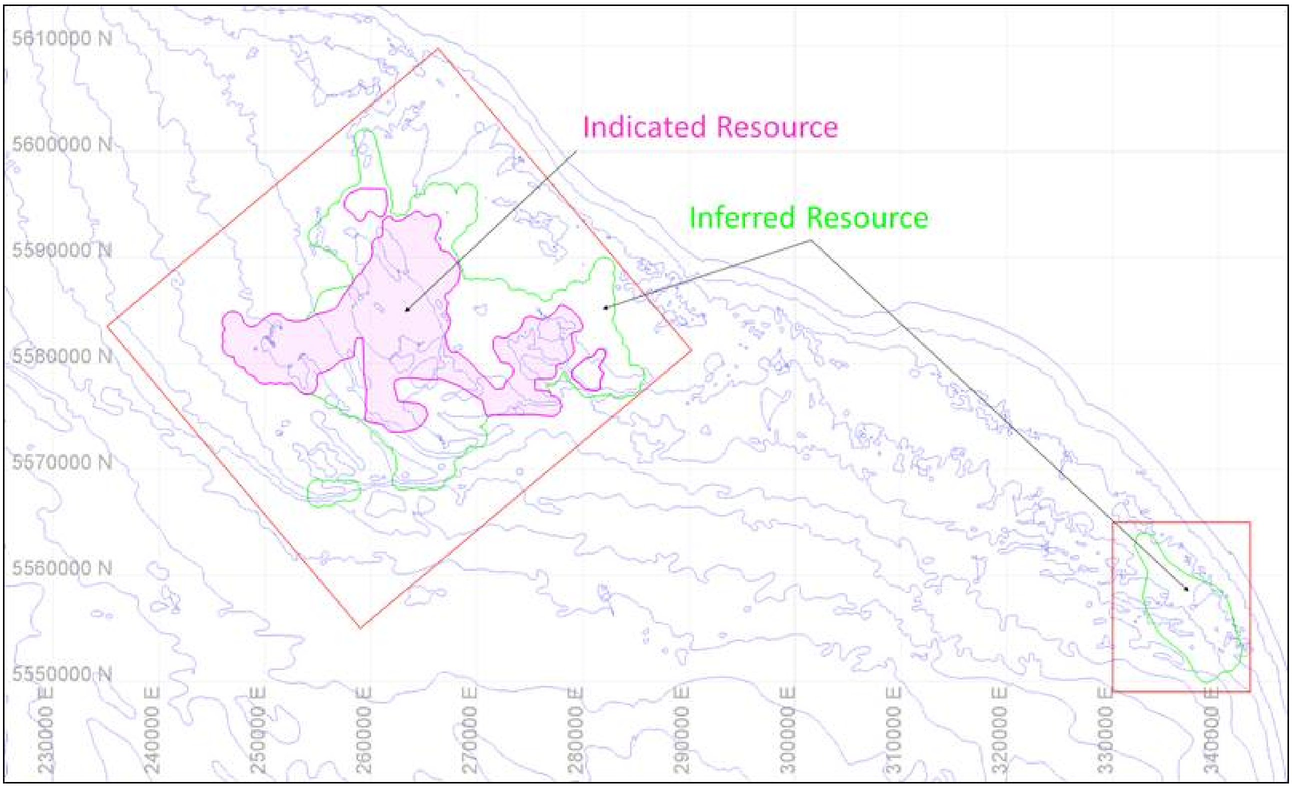


Figure 10-3: Resource Classification

For comparison to previous resource estimates Table 10-7 reports the models at a 5% Fe_2O_3 (head) cut-off grade.

Table 10-4: Tonnage and Head Grades (%) – Proposed Mine Area – 3.5% DTR Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	DTR*	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	1	165.4	11.31	7.37	11.07	53.57	1.15	10.64	1.12	5.41	0.20	0.22	2.68	94.55
	3	480.1	11.64	7.57	12.70	51.38	1.19	10.98	1.15	5.36	0.21	0.26	2.24	98.24
	6	304.0	9.71	5.88	13.11	53.15	1.00	11.01	1.18	4.80	0.18	0.24	2.65	96.06
	7	81.6	10.52	6.08	10.87	49.67	1.03	13.94	1.00	5.92	0.20	0.23	4.23	88.00
	9	3.9	8.26	4.66	14.16	53.64	0.82	11.04	1.23	4.48	0.17	0.23	2.59	98.38
Indicated Total		***1035.1***	***10.92***	***6.91***	***12.42***	***52.13***	***1.11***	***11.17***	***1.14***	***5.24***	***0.20***	***0.24***	***2.59***	***96.20***
Inferred	1	26.9	15.85	11.82	9.12	49.60	1.60	9.61	0.91	5.43	0.23	0.19	5.16	92.90
	6	5.2	10.07	5.79	13.37	51.47	1.04	11.67	1.13	5.22	0.20	0.26	2.32	95.12
	7	3.4	12.52	8.12	9.64	48.89	1.21	13.82	0.82	6.82	0.21	0.20	3.90	90.84
Inferred Total		***35.5***	***14.68***	***10.58***	***9.79***	***49.80***	***1.48***	***10.32***	***0.94***	***5.53***	***0.22***	***0.20***	***4.63***	***93.03***
Total		**1070.7**	**11.04**	**7.03**	**12.33**	**52.05**	**1.12**	**11.14**	**1.14**	**5.25**	**0.20**	**0.24**	**2.66**	**96.10**

*the DTR estimate is based on analytical DTR and calculated DTR values.

Table 10-5: Tonnage and Concentrate Grades (%) – Proposed Mine Area – 3.5% DTR Cut-Off Grade

Class	Domain	Mt	Fe	Al_2O_3	SiO_2	Ti	CaO	K_2O	MgO	Mn	P	LOI
Indicated	1	12.2	57.43	3.66	3.50	5.01	0.95	0.09	3.22	0.51	0.10	-3.17
	3	36.3	56.58	3.67	4.18	5.09	1.06	0.12	3.26	0.23	0.11	-3.04
	6	17.9	56.62	3.70	4.29	5.05	1.08	0.12	3.25	0.51	0.10	-3.07
	7	5.0	56.79	3.77	4.05	4.97	1.10	0.10	3.33	0.51	0.10	-3.12
	9	0.2	55.26	3.75	5.71	5.03	1.32	0.17	3.38	0.50	0.12	-2.93
Indicated Total		***71.5***	***56.73***	***3.68***	***4.10***	***5.06***	***1.05***	***0.12***	***3.26***	***0.37***	***0.10***	***-3.08***
Inferred	1	3.2	59.48	3.55	1.62	4.87	0.53	0.03	2.98	0.52	0.07	-3.38
	6	0.3	56.00	3.76	4.98	5.04	1.24	0.14	3.34	0.51	0.11	-3.07
	7	0.3	58.53	3.67	2.48	4.85	0.77	0.05	3.17	0.51	0.07	-3.27
Inferred Total		***3.8***	***59.06***	***3.58***	***2.02***	***4.88***	***0.62***	***0.05***	***3.03***	***0.52***	***0.08***	***-3.34***
Total		**75.3**	**56.83**	**3.67**	**4.02**	**5.06**	**1.03**	**0.11**	**3.25**	**0.37**	**0.10**	**-3.09**

*the DTR estimate is based on analytical DTR and calculated DTR values.

Table 10-6: Tonnage and Head Grades (%) – Outside Proposed Mine Area – 7.5% Fe_2O_3 Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	2	119.9	8.68	13.05	49.57	0.88	13.88	1.17	5.26	0.19	0.25	4.19	84.64
	3	56.3	9.23	14.02	51.32	0.93	12.14	1.19	5.25	0.19	0.26	2.50	92.76
	4	71.3	9.94	11.80	46.23	0.95	16.21	0.90	6.22	0.21	0.26	5.02	88.12
	5	37.3	9.11	14.17	50.43	0.91	12.68	1.21	5.65	0.20	0.27	2.31	84.36
	6	100.6	11.28	13.09	51.87	1.18	10.74	1.13	4.66	0.20	0.22	2.83	94.05
	7	282.1	8.92	13.73	51.09	0.89	12.61	1.21	5.34	0.20	0.24	2.73	89.14
	9	123.7	9.07	14.14	51.53	0.90	12.18	1.20	5.33	0.19	0.26	2.25	93.06
Indicated Total		***791.2***	***9.33***	***13.48***	***50.57***	***0.94***	***12.79***	***1.16***	***5.33***	***0.20***	***0.25***	***3.06***	***89.64***
Inferred	1	33.2	15.18	7.88	47.61	1.52	13.35	0.79	5.58	0.21	0.21	5.87	88.42
	2	171.7	8.49	14.40	50.33	0.87	12.64	1.29	4.82	0.18	0.24	3.50	86.74
	3	108.5	8.89	14.68	52.31	0.91	11.04	1.33	4.63	0.18	0.25	2.55	94.68
	4	93.3	8.87	11.21	45.66	0.85	17.48	0.90	6.11	0.20	0.23	6.35	83.62
	5	4.2	8.42	13.51	50.23	0.82	13.77	1.16	6.20	0.20	0.28	2.62	79.70
	6	279.6	11.17	12.16	51.43	1.13	11.55	1.08	5.13	0.20	0.22	3.06	94.30
	7	144.6	8.67	11.11	45.19	0.84	17.70	0.91	5.88	0.20	0.23	6.87	82.45
	8	60.6	9.08	9.12	54.13	0.78	12.62	0.85	6.82	0.18	0.14	4.50	90.51
	9	190.8	8.95	14.41	51.59	0.89	12.22	1.23	5.54	0.19	0.27	1.90	90.78
Inferred Total		***1086.5***	***9.59***	***12.65***	***50.08***	***0.95***	***13.24***	***1.10***	***5.40***	***0.19***	***0.23***	***3.83***	***89.58***
Total		**1877.7**	**9.48**	**13.00**	**50.29**	**0.95**	**13.05**	**1.13**	**5.37**	**0.19**	**0.24**	**3.50**	**89.60**

Table 10-7: Tonnage and Head Grades (%) – Full Area Reported – 5% Fe_2O_3 Cut-Off Grade

Class	Domain	Mt	Fe_2O_3	Al_2O_3	SiO_2	TiO_2	CaO	K_2O	MgO	MnO	P_2O_5	LOI	REC
Indicated	1	213.5	10.25	11.18	54.86	1.04	10.44	1.16	5.08	0.18	0.20	2.92	93.45
	2	330.3	7.34	13.55	50.28	0.76	13.56	1.29	4.16	0.15	0.23	5.49	87.50
	3	617.8	10.74	13.20	52.20	1.10	10.70	1.22	5.04	0.19	0.25	2.32	97.75
	4	77.4	9.71	11.96	46.48	0.93	16.11	0.92	6.14	0.20	0.26	4.98	88.58
	5	110.8	7.27	14.87	52.35	0.77	11.48	1.42	3.91	0.15	0.24	4.05	90.65
	6	606.2	8.92	13.21	55.07	0.93	10.13	1.26	4.34	0.17	0.22	2.70	95.68
	7	569.8	8.36	14.07	52.39	0.85	11.76	1.31	4.72	0.18	0.23	2.95	90.24
	9	157.3	8.64	14.40	52.08	0.87	11.87	1.24	5.01	0.18	0.25	2.37	93.92
Indicated Total		***2683.0***	***9.07***	***13.37***	***52.70***	***0.93***	***11.39***	***1.25***	***4.69***	***0.18***	***0.23***	***3.13***	***93.30***
Inferred	1	64.0	14.93	8.54	48.83	1.50	11.76	0.87	5.37	0.21	0.20	5.73	90.00
	2	340.0	7.61	15.30	50.84	0.80	12.19	1.40	3.98	0.16	0.23	4.00	87.91
	3	179.9	8.11	15.37	53.85	0.85	10.05	1.46	4.04	0.16	0.24	2.44	95.79
	4	173.1	7.84	11.10	44.57	0.75	18.59	0.90	5.52	0.18	0.22	8.08	83.78
	5	7.5	7.57	13.45	52.20	0.77	12.61	1.28	5.19	0.17	0.26	3.34	83.64
	6	377.0	9.97	12.76	53.14	1.03	10.64	1.21	4.60	0.18	0.21	3.18	94.70
	7	315.7	7.52	12.48	48.00	0.75	15.72	1.12	4.76	0.17	0.22	6.43	85.56
	8	191.6	7.04	9.68	56.77	0.64	11.65	1.01	5.11	0.14	0.14	5.45	90.83
	9	529.7	7.38	16.21	52.95	0.76	11.10	1.45	4.17	0.16	0.25	2.18	91.57
Inferred Total		***2178.6***	***8.17***	***13.64***	***51.56***	***0.83***	***12.44***	***1.25***	***4.52***	***0.17***	***0.22***	***4.14***	***90.26***
Total		**4861.6**	**8.67**	**13.50**	**52.19**	**0.89**	**11.86**	**1.25**	**4.62**	**0.17**	**0.23**	**3.58**	**91.94**

10.5 Comparison to the 2012 Model

The 2013 mineral resource model incorporated a number of changes from the 2012 model. These changes were applied to the “Area 2” model over the proposed mine area. The Koitiata model remains unchanged from 2012. In summary the changes were:

- Bathymetry – The bathymetric surface was updated to include more detailed data acquired with WASSP® multi beam sonar data.
- Database
  - 103 records removed – grab samples, bulk samples, drill holes
  - 42 drill holes from the 2012 database were removed for the 2013 modelling. Eight (8) of these drill holes were used in the 2012 resource estimation.
  - 158 drill holes completed in 2013 were added to the database.
- The base of mineralisation (BOM) was revised with the 2013 drilling.
- Model rotated clockwise 50° around 259000 E, 5555000 N to align the blocks with the mining direction.
- The model Parent Block size was changed from 500 m × 500 m to 300 m × 300 m to reflect the expected Selective Mining Unit (SMU) size.
- Variography was revised or reviewed with new data.

The following model variables are unchanged

- Estimation domains.
- Kriging plan (except variography).
- The proposed Mine Area.
- Top and Bottom RL of model.
- Reporting criteria and constraints.

The impact of each of the parameter changes were individually assessed. Several additional models were constructed and estimated to define the effect of changing block sizes and orientation on the 2102 and 2013 models. The principal means of comparison was by Grade-Tonnage curves. The drill hole data used in each model was compared statistically and visually. The impact of each of the changes is listed in Table 10-8. The impact of the change in the base of mineralisation (BOM) is measured, the remainder are estimated.

Table 10-8: Impact of Model Changes

Bathymetry/BOM	+4% volume
Rotation	<1%
Block Size	- tonnes :10% @3.5% cut-off + grade :2% @3.5% cut-off
Database	+/- tonnes :+4% @3.5% cut-off - grade :6% @3.5% cut-off
Variography	<1%

The change in tonnes and grade with block size is as expected. Smaller blocks give better selectivity with a resulting decrease in tonnes and increase in grade. The decrease in tonnes is somewhat masked by the overall increased volume due to the recent deeper drilling.

The most significant difference between the 2012 and 2013 models is the drill data. Infill drilling in the mine area has shown the deposit to be less homogenous than expected. The infill drilling has constrained the higher grade areas more than previously and resulted in a drop in the average head grade of the mineral resource.

Appendix I contains more detail of the models comparison and validation.

10.6 Compliance with the JORC Code Assessment Criteria

The JORC Code (2012) describes a number of criteria, which must be addressed in the documentation of Mineral Resource estimates, prior to public release of the information. These criteria provide a means of assessing whether or not the data inventory used in the estimate is adequate for that purpose. The resource estimate stated in this document was based on the criteria set out in Table 1 of that Code. These criteria have been discussed in the main body of the document and are summarised below. The JORC Code Assessment Criteria in the following table are italicised.

Table 10-9: JORC Code (2012)

Criteria	JORC Code Explanation	Commentary
Section 1 Sampling Techniques and Data (Criteria in this section apply to all succeeding sections.)		
Sampling techniques	■ *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.* ■ *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* ■ *Aspects of the determination of mineralisation that are Material to the Public Report.* ■ *In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	■ The material being sampled is subsea sand originally deposited in marine and terrestrial environments. ■ Samples used in the resource estimation are from drill holes only. Grab samples have only been used as qualitative indicators of the presence of magnetic heavy minerals during early exploration. ■ Drilling used a passive triple tube reverse circulation system. The full sample for each metre was collected and a sub-sample split from this for analysis by XRF. ■ Drill samples from the proposed mine area have been subject to Davis Tube Recovery to determine the magnetically recoverable portion of the sample. The concentrate recovered has been analysed by XRF.

Criteria	JORC Code Explanation	Commentary
Drilling techniques	■ *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	■ The drill sampling uses a proprietary passive triple tube reverse circulation technique drilling a 75.75 mm diameter hole to a maximum depth of 11 m. ■ Six 5 inch diameter RC drill holes were drilled in 2012 to a maximum depth of 30 m.
Drill sample recovery	■ *Method of recording and assessing core and chip sample recoveries and results assessed.* ■ *Measures taken to maximise sample recovery and ensure representative nature of the samples.* ■ *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	■ Golder has reviewed the drilling and sampling and considers that a representative sample is being collected. Sample weights are recorded. Oversized samples due to hole 'blow outs' are excluded from the resource estimation.
Logging	■ *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* ■ *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.* ■ *The total length and percentage of the relevant intersections logged.*	■ The qualitative logging of samples is of sufficient detail to support the current mineral resource. ■ All resource drill holes have been logged.
Sub-sampling techniques and sample preparation	■ *If core, whether cut or sawn and whether quarter, half or all core taken.* ■ *If non-core, whether riffled, tube sampled, rotary split, etc., and whether sampled wet or dry.* ■ *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* ■ *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* ■ *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* ■ *Whether sample sizes are appropriate to the grain size of the material being sampled.*	■ 1 m samples were taken from the sample cyclone. The dried sample is split using a rotary splitter. Sample sizes are appropriate for the sandy material being collected. ■ Duplicate samples are routinely submitted to monitor the sample preparation process. ■ All procedures are well documented and understood by the operational personnel.

Criteria	JORC Code Explanation	Commentary
Quality of assay data and laboratory tests	■ *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* ■ *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* ■ *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	■ The analytical techniques, particularly the Davis Tube Recovery analysis, are appropriate for this type of deposit. ■ Regular reference standards (IRM), blanks and duplicate samples are submitted to the laboratory to monitor the accuracy and precision of the analysis process and results. In addition referee samples are sent to alternate laboratories. ■ Analysis of the QAQC sample results to date indicate that the accuracy and precision of the assay data is adequate for the mineral resource estimation.
Verification of sampling and assaying	■ *The verification of significant intersections by either independent or alternative company personnel.* ■ *The use of twinned holes.* ■ *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* ■ *Discuss any adjustment to assay data.*	■ Independent verification of sampling has not been undertaken due to the logistics involved. ■ At Golders request a series of samples from the 2010 drilling campaign were resubmitted to an alternative laboratory. These referee samples returned analysis results consistent with the original analysis. ■ Drilling and sampling of several holes has been observed by Golder. Referee sampling has been used to validate the accuracy and precision of historical samples. Twin holes have been drilled but the results from twin holes are inconclusive. ■ All sampling and data management procedures are documented ■ Data management is considered adequate. ■ Rotary Reverse circulation sampling has been trialled. Golder observed the drilling of two of these holes and considers the samples to be non-representative due to losses. Data from these holes has not been used.
Location of data points	■ *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* ■ *Specification of the grid system used.*	■ For the scale of the deposit and at this stage of development the location of samples by hand held GPS is considered adequate. ■ GPS data is in latitude and longitude. ■ Modelling data is in UTM – WGS 84 Zone 60

Criteria	JORC Code Explanation	Commentary
	■ *Quality and adequacy of topographic control.*	■ Commercial/Public domain bathymetric data is considered adequate over most of the tenements and good in the mine area where the data has been supplemented with WASSP multibeam sonar data and multibeam survey data acquired by NIWA.
Data spacing and distribution	■ *Data spacing for reporting of Exploration Results.* ■ *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* ■ *Whether sample compositing has been applied.*	■ Much of the resource area is now drilled on a nominal 1000 m by 1000 m grid. Analysis to date suggests that this is an adequate sample spacing to define an Indicated Mineral Resource. Deeper drilling may start to introduce more variability and lead to a requirement for infill drilling. ■ Samples are not composited for analysis
Orientation of data in relation to geological structure	■ *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* ■ *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	■ All drill holes are vertical providing the optimum orientation for sampling these bedded sand deposits.
Sample security	■ *The measures taken to ensure sample security.*	■ Sample security is good with all samples being under TTR supervision up until submission at the laboratory. ■ Laboratory chain of custody and security have been reviewed by Golder and are considered fit for purpose.
Audits or reviews	■ *The results of any audits or reviews of sampling techniques and data.*	■ In 2010 Golder undertook a detailed audit of the drill hole database. Minor anomalies in the database were found and corrected. ■ In 2012 QG (Perth) undertook a due diligence of the resource data and estimation. Golder has not been advised of the conclusions and recommendations from QG's work.

Criteria	JORC Code Explanation	Commentary
		■ To address issues raised by Golder in their QAQC data analysis, Jeremy Batchelor of Chem Tek Consulting undertook an independent lab audit and QAQC data analysis in 2013 finding the laboratory procedures and results satisfactory.
Section 2 Reporting of Exploration Results (Criteria listed in the preceding section also apply to this section.)		
Mineral tenement and land tenure status	■ *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* ■ *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	■ TTRL hold granted Continental Shelf Licence 50753 and Exploration Permit 54068, due for renewal in December 2014 and December 2017 respectively. TTRL have current submissions for Exploration Permits 54270, 54271 and 54272 which will be valid for five years when granted. ■ TTRL have a current submission for Mining Permit 55581. The submission has applied for a 20 year period. ■ All tenements are owned 100% by TTRL. ■ Royalty commitment for mining permit 55581 will the higher of ■ an ad valorem royalty of 2% of the net sales revenue of the minerals obtained under the permit; and ■ an accounting profits royalty of 10% of the accounting profits, or provisional accounting profits, as the case may be, of the minerals obtained under the permit. ■ The mining permits are subject to: ■ Marine consents under the Exclusive Economic Zone and Continental Shelf (Environmental Effects) Act 2012 (EEZA) for activities beyond the 12 nm limit. ■ Resource consents under the Resource Management Act 1991 (RMA) for activities (including discharges) within the 12 nm limit. ■ Marine discharge consents under the EEZA or Discharge Management Plans under the Maritime Transport Act 1994 (MTA) for discharges beyond the 12 nm limit.

Criteria	JORC Code Explanation	Commentary
		■ Mining permit applications require consultation with any iwi or hapu whose rohe may be directly affected by the permit.
Exploration done by other parties	■ *Acknowledgment and appraisal of exploration by other parties.*	■ Some petroleum bore logs record near surface ironsands ■ Geophysical surveys were largely reconnaissance in nature providing limited offshore detail. ■ Limited, historical sampling of shallow offshore deposits has been undertaken providing indicative results only.
Geology	■ *Deposit type, geological setting and style of mineralisation.*	■ The deposit is a submarine aeolian/alluvial/marine accumulation of ironsand in palaeo channels, beaches and dunes. The main mineral of interest is titano magnetite.
Drill hole Information	■ *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ▪ *easting and northing of the drill hole collar* ▪ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ▪ *dip and azimuth of the hole* ▪ *down hole length and interception depth* ▪ *hole length.* ■ *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	■ 799 vertical seafloor drill holes have been drilled. ■ The current resource uses 633 of these drill holes, drilled and sampled, averaging 5.7 m in depth for a total of 3604.5 m. ■ The remaining holes are reconnaissance, bulk sampling and trial holes.
Data aggregation methods	■ *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	■ Exploration drilling results are not reported here.

Criteria	JORC Code Explanation	Commentary
	■ *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* ■ *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	
Relationship between mineralisation widths and intercept lengths	■ *These relationships are particularly important in the reporting of Exploration Results.* ■ *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* ■ *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	■ The ironsands are bedded deposits. Drilling to date has only defined the true thickness of the deposit in six drill holes.
Diagrams	■ *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	■ See figures 1 to 6.
Balanced reporting	■ *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	■ Exploration results are not reported here.
Other substantive exploration data	■ *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	■ Exploration data to date includes geophysical surveys, grab samples, bulk samples and drilling. Some metallurgical test work has been done on magnetic and gravity recovery including the construction of a pilot plant. Enough data is available to make a reasonably confident estimate of the dry bulk density.
Further work	■ *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling).*	■ Further infill drilling is planned to extend the available mining area.

Criteria	JORC Code Explanation	Commentary
	■ *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	■ Pending budget approval a detailed marine vessel based geophysical magnetic survey over the mine area is planned.
Section 3 Estimation and Reporting of Mineral Resources (Criteria listed in section 1, and where relevant in section 2, also apply to this section.)		
Database integrity	■ *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* ■ *Data validation procedures used.*	■ Golder has undertaken a detailed audit of the drill hole database validating the data and ensuring that adequate security and backup procedures are in place. ■ Drill data is routinely checked for internal consistency, anomalies and omissions prior to each resource estimation.
Site visits	■ *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* ■ *If no site visits have been undertaken indicate why this is the case.*	■ The site has been visit by the competent person, Stephen Godfrey, on three occasions. 1) January, 2010 – reviewed drilling and sampling. Recommendations for improved procedures made and implemented. 2) July 2012 – reviewed pilot plant, project in general 3) February 2013 – reviewed rotary RC drilling. Identified sampling issues.
Geological interpretation	■ *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* ■ *Nature of the data used and of any assumptions made.* ■ *The effect, if any, of alternative interpretations on Mineral Resource estimation.* ■ *The use of geology in guiding and controlling Mineral Resource estimation.* ■ *The factors affecting continuity both of grade and geology.*	■ Preliminary drilling showed the deposit to be relatively consistent in the top six metres with most material being mineralised. ■ The 2013 infill drilling is now showing better qualitative correlation with the airborne magnetic surveys with higher grade mineralisation in general being coincident with magnetic highs. The correlation is not always consistent and the impact on exploration and the resource is still being assessed. ■ Confidence in the geological interpretation is medium to high.
Dimensions	■ *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	■ The deposit has been drilled over a strike length of 100 km and a width of 6 to 12 km.

Criteria	JORC Code Explanation	Commentary
Estimation and modelling techniques	■ *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* ■ *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* ■ *The assumptions made regarding recovery of by-products.* ■ *Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulfur for acid mine drainage characterisation).* ■ *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* ■ *Any assumptions behind modelling of selective mining units.* ■ *Any assumptions about correlation between variables.* ■ *Description of how the geological interpretation was used to control the resource estimates.* ■ *Discussion of basis for using or not using grade cutting or capping.* ■ *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	■ The available sampling data is sufficient to allow variogram models and kriging parameters to be defined. The models were estimated using Ordinary Kriging. The estimation has a maximum extrapolation of 1000 m from any data point. ■ The models were constructed using Maptek's Envisage software ('Vulcan") and estimated using proprietary Golder software based on the GSLIB geostatistical library. ■ The estimate has been made into 300 m × 300 m × 1 m parent blocks oriented at 050°. These blocks represent the mining SMU as defined in the PFS, and are approximately one third of the average drill spacing. ■ Head Fe_2O_3 and DTR show a positive correlation. This correlation has been used to estimate DTR outside the mining area where DTR has been measured. ■ The sample population showed no/limited extreme outlier so no grade cutting or grade restraint was applied. ■ The estimation was unfolded to the bathymetric surface. ■ The models have estimated the major and deleterious elements for the -2 mm fraction for the full model. In addition Davis Tube Recovery and concentrate grades have been estimated for the proposed mining area. ■ The models were validated against the drill holes visually and statistically. The estimations for both models are considered to have a medium to high level of confidence.
Moisture	■ *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	■ All tonnages are estimated on dry basis consistent with the sample analysis which is reported as a dry mass percent.

Criteria	JORC Code Explanation	Commentary
Cut-off parameters	■ *The basis of the adopted cut-off grade(s) or quality parameters applied.*	■ The Fe_2O_3 cut-off used to define the mineralisation was based on the population statistics for Fe_2O_3. The DTR cut-off of 3.5% applied to reporting is based on preliminary economic estimates of mining cut-off grade. Based on the good correlation between head Fe (or Fe_2O_3) and DTR 3.5% DTR is equivalent to 7.5% Fe_2O_3.
Mining factors or assumptions	■ *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	■ The current assumption is that this will be a dredging operation. It will be a bulk mining scenario with any subgrade overburden incorporated into the mineralised zone where practicable. ■ Consequently only a base of mineralisation is defined in the geological model with minor amounts of subgrade overburden and interburden incorporated into the model. ■ The base of mineralisation was defined at 4% Head Fe_2O_3 based on the population statistics of the analyte. DTR analyses are incomplete for the entire model area and could not be used to define the cut off, however there is a strong positive correlation between Fe_2O_3 and DTR.
Metallurgical factors or assumptions	■ *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	■ No metallurgical recovery factors have been applied. Samples are screened at 2 mm before analysis. The screened recovery used to weight the head grade estimation. Davis Tube Recovery (DTR) analyses have been performed on the samples for 167 drill holes in the proposed mining area. Where DTR analyses were not available a calculated DTR value based on the relationship between Head Fe_2O_3 and analytical DTR was used. The DTR and screened recovery have been used to weight the concentrate grade estimation.

Criteria	JORC Code Explanation	Commentary
Environmental factors or assumptions	■ *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	■ Tailings form the mining operation are to be returned to the seafloor in mined out areas. ■ Baseline environmental studies are in place and have determined no deleterious impact to date. ■ TTRL submitted an environmental consent application to the EPA on 21 October 2013.
Bulk density	■ *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* ■ *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit.* ■ *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	■ Dry bulk density was determined by laboratory analysis and verified by comparison to the theoretical bulk density. Bulk density is sensitive to the heavy mineral content. A regression formula was used to estimate bulk density based on the Fe content. ■ A small number of samples (3) suggest decreasing porosity with Fe grade. If the samples prove valid they have the potential to increase the tonnage of the deposit by several percent.
Classification	■ *The basis for the classification of the Mineral Resources into varying confidence categories.* ■ *Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* ■ *Whether the result appropriately reflects the Competent Person's view of the deposit.*	■ Those parts of the resource classified as Indicated have been sampled at density considered adequate to support the classification. No adverse quality or geological uncertainty parameters affect this classification. The Inferred classification of the deposit reflects the assumed geological and geostatistical continuity in parts of the current model where the drill spacing exceeds 1000 m by 1000 m. ■ Classification of the deposit was undertaken by the competent person.

Criteria	JORC Code Explanation	Commentary
Audits or reviews	■ *The results of any audits or reviews of Mineral Resource estimates.*	■ The current mineral resource estimate has not been externally audited. In 2012 QG (Perth) undertook a due diligence of the resource data and estimation.
Discussion of relative accuracy/ confidence	■ *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* ■ *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* ■ *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	■ The current resource is a global estimate. The relatively sparse data does not allow a high confidence local estimate. ■ The model is considered adequate to use in a mine planning study for a bulk dredging style operation.

11.0 CONCLUSIONS AND RECOMMENDATIONS

The current drilling and sampling program adequately defines the resource to a depth of 6 m to 9 m for most of the deposit. The data quality and quantity supports Indicated and Inferred resource classifications applied.

The current shallow drilling campaign is increasing the size of the resource; however, increased drill depth will potentially increase the resource tonnes within a smaller area.

There is a qualitative correlation between the sample grades and the geophysical model becoming more apparent with the increased data density. This relationship should be investigated further as an aid to exploration and, potentially, to corroborate the Fugro geophysical “resource”. If a correlation between the drill sampling and the geophysical response can be established there may be potential to regard this more formally as an Exploration Target or (with confirmatory drill sampling) as a Mineral Resource.

12.0 LIMITATIONS

Your attention is drawn to the document “Limitations”, which is included in Appendix I of this report. The statements presented in this document are intended to advise you of what your realistic expectations of this report should be, and to present you with recommendations on how to minimise the risks associated with this project. The document is not intended to reduce the level of responsibility accepted by Golder Associates, but rather to ensure that all parties who may rely on this report are aware of the responsibilities each assumes in so doing.

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Report Signature Page

GOLDER ASSOCIATES PTY LTD

Stephen Godfrey
Associate, Principal Resource Geologist

Sia Khosrowshahi
Principal

SAG/SK/hsl

A.B.N. 64 006 107 857

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