THE MSA GROUP

Specialist Consultants to the Mining Industry

Zimbabwe Lithium Company
Kamativi Lithium Tailings Project
Matabeleland North Province, Zimbabwe

NI 43-101 Technical Report

Prepared By The MSA Group (Pty) Ltd for:

Zimbabwe Lithium Company (Mauritis)

Prepared By:

Michael Cronwright	M.Sc., Pr. Sci. Nat., FGSSA
Ipelo Gasela	M.Sc. (Eng), Pr. Sci. Nat., MGSSA
John Derbyshire	B.Sc. Eng. (Chem), Pr.Eng., FSAIMM

Effective Date: 10 September 2018
Report Date: 05 November 2018

MSA Project No.: 3720

The MSA Group (Pty) Ltd | Registration No: 2000/002800/07
Tel: +27 (0)11 880 4209 | Fax: +27 (0)11 880 2184 | email: info@msagroupservices.com
Henley House, Greenacres Office Park: Cnr Victory and Rustenburg Roads, Victory Park, 2195
PO Box 81356, Parkhurst, 2120, South Africa
Directors: BJ Burnand, KD Scott, NN Buthelezi, SZ Majola

www.msagroupservices.com

IMPORTANT NOTICE

This report was prepared as a National Instrument NI 43-101 Technical Report for Zimbabwe Lithium Company (Mauritius) ("ZIM") by The MSA Group (Pty) Ltd (MSA), South Africa. The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in MSA's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by ZIM subject to the terms and conditions of its contract with MSA. Except for the purposes legislated under Canadian provincial securities law, any other uses of this report by any third party is at that party's sole risk.

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Henley House, Greenacres Office Park:
Cnr Victory and Rustenburg Roads, Victory Park, 2195
PO Box 81356, Parkhurst, 2120, South Africa
Directors: BJ Burnand, KD Scott, NN Buthelezi, SZ Majola

CERTIFICATE OF QUALIFIED PERSON

I, Michael Cronwright, Pr. Sci. Nat. do hereby certify that:

1. I am Principal Consultant of:

   The MSA Group (Pty) Ltd
   Henley House, Greenacres Office Park
   Cnr Rustenburg and Victory Roads
   Victory Park, Gauteng, South Africa,
   2196

2. This certificate applies to the technical report titled "Zimbabwe Lithium Company, Kamativi Lithium Tailings Project, Matabeleland North Province, Zimbabwe, NI 43-101 Technical Report, that has an effective date of 10 September 2018 and a report date of 05 November 2018 (the Technical Report).
3. I graduated with a B.Sc. (Hons) degree in Geology from the University of Natal (Durban) in 1998. In addition, I have obtained a M.Sc. in Exploration Geology from Rhodes University on 2005.
4. I am a Professional Natural Scientist (Geological Science) with the South African Council for Natural Scientific Professions (SACNASP) and a fellow of the Geological Society of South Africa.
5. I have worked as a geologist for a total of 18 years, during which time I have worked in a number of roles; as a scientific officer at the Council for Geoscience; as middle and senior management for a geological consultancy and executed exploration projects, conducted reviews and audits on numerous projects covering a variety of commodities and mineralisation styles, including pegmatite hosted lithium, tin and coltan projects.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I visited the Kamativi Lithium Tailings Project on 7-8 August 2017 for 2 days and on 23-24 April 2018
8. I am responsible for, or co-responsible for, the preparation of sections 1-12, 19, 20 and 24-27 of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer according to the definition of independence described in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1 and, as of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 5th day of November 2018.

"Signed and Stamped"

Michael Cronwright, Pr. Sci. Nat., FGSSA

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The MSA Group (Pty) Ltd
Registration No: 2000/002800/07
Tel: +27 (0)11 880 4209 Fax: +27 (0)11 880 2184
email: info@msagroupservices.com
Henley House, Greenacres Office Park:
Cnr Victory and Rustenburg Roads, Victory Park, 2195
PO Box 81356, Parkhurst, 2120, South Africa
Directors: BJ Burnand, KD Scott, NN Buthelezi, SZ Majola

I, Ipelo Gasela, Pr. Sci. Nat. do hereby certify that:

1. I am Senior Mineral Resource Consultant of:

   The MSA Group (Pty) Ltd
   Henley House, Greenacres Office Park
   Cnr Rustenburg and Victory Roads
   Victory Park, Gauteng, South Africa,
   2196

2. This certificate applies to the technical report titled "Zimbabwe Lithium Company, Kamativi Lithium Tailings Project, Matabeleland North Province, Zimbabwe, NI 43-101 Technical Report, that has an effective date of 10 September 2018 and a report date of 05 November 2018 (the Technical Report).
3. I graduated with a B.Sc. (Hons) degree in Geology from the University of the Witwatersrand in 2004. In addition, I have obtained a M.Sc. (Eng) from the University of the Witwatersrand in 2018.
4. I am a Registered Member of The South African Council for Natural Scientific Professions (SACNASP) and a Member of the Geological Society of South Africa (GSSA).
5. I have worked as a geologist for a total of 13 years, during which time I have worked in a number of roles; as an Evaluation Management trainee and a Senior Evaluator at a mine and corporate office for a mining company and a Mineral Resource consultant for mining consultancies, where I have undertaken Mineral Resource estimates, due diligence reviews and audits for a variety of commodities.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited the Kamativi Lithium Tailings Project.
8. I am responsible for, or co-responsible for, the preparation of sections 1, 14 and 25-27 of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer according to the definition of independence described in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1 and, as of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 5th day of November 2018.

"Signed and Stamped"

Ipelo Gasela, Pr. Sci. Nat., MGSSA

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PO Box 81356, Parkhurst, 2120, South Africa
Directors: BJ Burnand, KD Scott, NN Buthelezi, SZ Majola

CERTIFICATE OF QUALIFIED PERSON

I, John Derbyshire, Pr. Eng. do hereby certify that:

1. I am an Associate Metallurgist of:

   The MSA Group (Pty) Ltd
   Henley House, Greenacres Office Park
   Cnr Rustenburg and Victory Roads
   Victory Park, Gauteng, South Africa,
   2196

2. This certificate applies to the technical report titled "Zimbabwe Lithium Company, Kamativi Lithium Tailings Project, Matabeleland North Province, Zimbabwe, NI 43-101 Technical Report, that has an effective date of 10 September 2018 and a report date of 05 November 2018 (the Technical Report).
3. I graduated with a B.Sc. Eng (Chem) degree from the University of Witwatersrand in 1981.
4. I am a registered Professional Engineer (Pr.Eng.) with the Engineering Council of South Africa and a Fellow of the South African Institute of Mining and Metallurgy (FSAIMM).
5. I have worked as a metallurgist for a total of 37 years, with plant and operational experience in senior positions in the South African mining industry and covering a range of commodities.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited the Kamativi Lithium Tailings Project.
8. I am responsible for, or co-responsible for, the preparation of sections 1, 13, and 25-27 of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
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Dated this 5th day of November 2018.

"Signed and Stamped"

John Derbyshire, Pr. Eng., FSAIMM

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1 SUMMARY

The MSA Group (Pty) Ltd ("MSA") has been commissioned by Zimbabwe Lithium Company ("ZIM"/ "the Company") to provide an Independent Technical Report on the Company's exploration activity on the Kamativi Lithium Tailings Property ("the Property") located in Matabeleland North, Zimbabwe. The Kamativi Lithium Tailings Project ("Kamativi tailings" / "Kamativi Project" / "the Project") comprises the tailings dump derived from the historical tin mining conducted within ML No. 12 at the now disused Kamativi Tin Mine

The Technical Report has been prepared in accordance with the disclosure and reporting requirements set forth in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP, Form 43-101F1, and the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on May 10, 2014.

1.1 Ownership

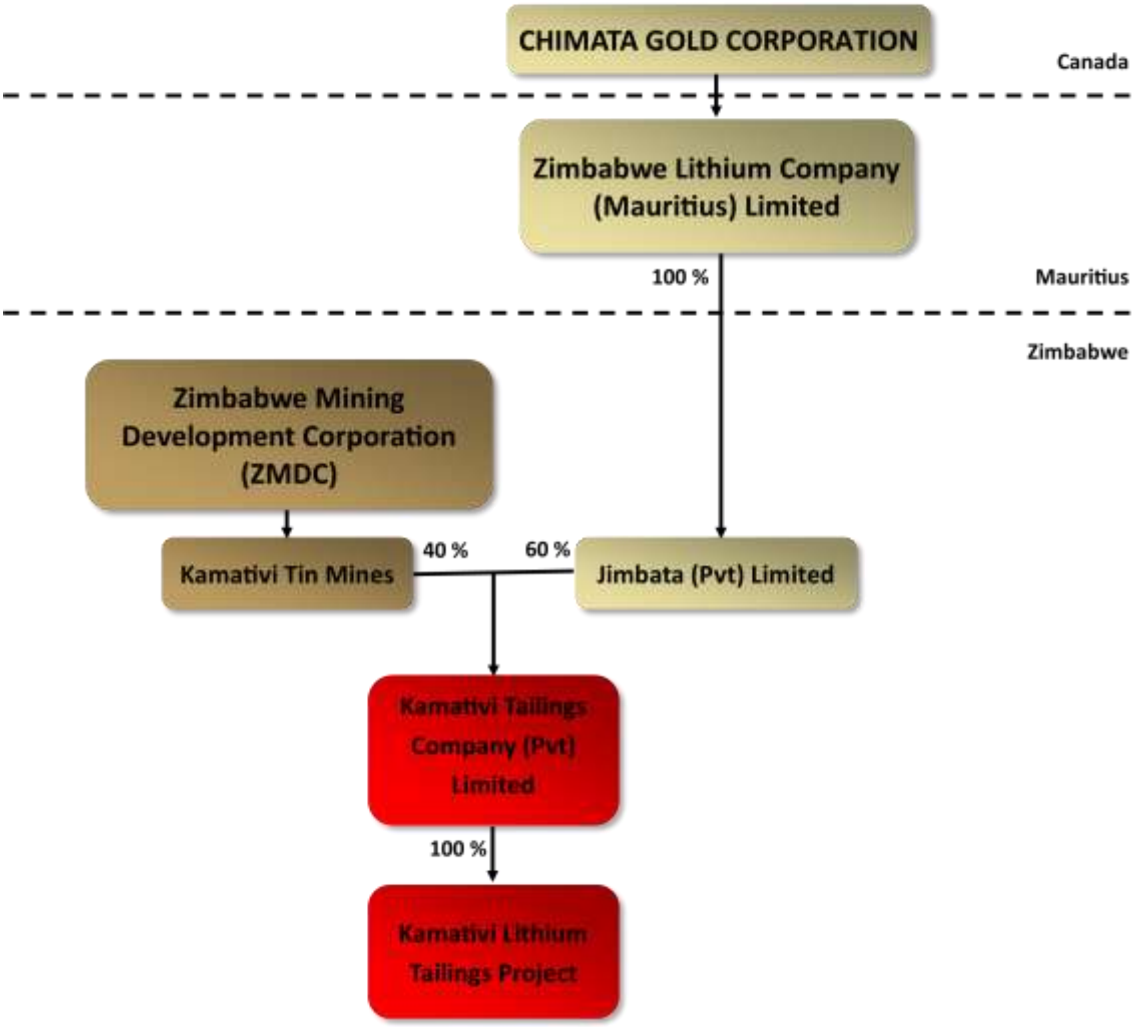
On 14 February 2018, Chimata Gold Corporation ("Chimata") announced the company had entered into a binding letter of intent with Zimbabwe Lithium Company (Mauritius) Limited ("ZIM"), a privately held company incorporated under the laws of Mauritius (Chimata, 2018). In terms of the Letter of Intent, Chimata will subscribe to share capital of ZIM for an initial subscription of 19% of ZIM's share capital in exchange for the allocation by Chimata of an amount of shares representing 19 % of its then outstanding share capital to ZIM. Chimata has the right to further acquire the remaining issued and outstanding share capital of ZIM upon fulfilling certain terms and conditions as set out in the Letter of Intent, the whole resulting in ZIM becoming a subsidiary of Chimata.

The Kamativi project is a joint venture ("JV") between the Zimbabwe Mining Development Corporation ("ZMDC"), owners of Kamativi Tin Mines which holds 40 % of the Project, and Jimbata (Pvt) Ltd ("Jimbata"), which holds 60 %. A JV Agreement was entered into between Lintmar (Private) Limited ("Lintmar") and ZMDC on 2 February 2018. A letter from Jimbata, dated 16 February 2018 confirms the cession by Lintmar rights and interests in the Kamativi Mine Tailings Dump to Jimbata, including all aspects of the JV Agreement with ZMDC.

The JV Company (Jimbata (60 %) and Kamativi Tin Mines (40 %), Kamativi Tailings Company (Pvt) Limited, was incorporated on 16 February 2018 as per the Companies Act [Chapter 24:03] of Zimbabwe.

On positive outcome of this NI 43-101 Technical Report, Chimata and ZIM will enter into a Definitive Agreement which gives Chimata the right, on completion of a Mineral Resource estimate undertaken in accordance with NI 43-101, and the fulfilment of the requirements as set out in the Letter of Intent, to acquire 100 % equity ownership of ZIM. This would give Chimata ownership over ZIM's equity interests. ZIM is a 100 % equity owner of Jimbata. The ownership structure of the Project is shown in Figure 1-1.

Figure 1-1
Project ownership structure



1.2 Property Description and Location

The Kamativi Project is located outside the village of Kamativi in the Matabeleland North Province of Zimbabwe. The Project, which is identified as a lithium-bearing tailings dump deposit associated with the disused Kamativi tin mine, is located approximately 185 kilometres east-south-east of Victoria Falls, approximately 84 km by tar road east of Hwange and approximately 310 km northwest of Bulawayo.

The Project comprises an area contained within the Kamativi Mining Lease No. 12 (“ML No. 12”) which is held by the Kamativi Tailings Company (Pvt) Limited.

1.3 Exploration and Mining History

The first reports of tin in the region date back to about 1920. The exploration and mining history are summarised in Table 1-1.

Table 1-1
Exploration and Mining History

Year	Description
~ 1920	Initial reports of the occurrence of tin
1920 - 1935	First pegging of Claims by Mr R.H. Aldworth (1935)
1936 - 1944	Production of cassiterite concentrates from alluvial deposits and surface rubble (panning and hydro-sluicing). First production declared by Messrs. R.H. Aldworth and S. Sauerman
1939-1949	Increase in production due to erection of small mills and the construction of a smelter in Bulawayo Tributors under the Government Ex-Serviceman Rehabilitation Scheme worked the claims (selective underground mining) after World War II
1949-1951	Rhodoak Limited set up the first Kamativi Tin Mines Limited
1951	Kamativi Tin Mines Limited acquired by Oakes Trust
1952	N.V. Billiton Maatschappij, a Dutch Naamloze Vennootschap ("N.V.") or public company, assumed financial and technical control A 400 tonnes per day pilot plant and an oil-fired drum-type rotary kiln were installed By 1964, an average of 64,000 tonnes of ore was being processed Production extended to other products (tin solder and bearing metals
1970-1986	Industrial Development Corporation of Zimbabwe ("IDC") acquired a controlling interest and ultimately entire shareholding Mill capacity increased to 2,000 tonnes per day, producing 950 tonnes of refined tin and alloys per year and approximately 23 tonnes of tantalite (Ta_2O_5) in slags. The plants processing capacity increased from 56,000 tonnes per month to 80,000 tonnes per month during this period
1986-1994	IDC transferred 91.3 % shares in Kamativi Tin Mines Limited to the ZMDC Economic reserves became depleted and tin production decreased to 1,060 tonnes in 1991 and 800 tonnes in 1993.
1994	Closure of the mine due to low tin prices, following the tin price crash of 1985, coupled with falling ore grades

By 1977 approximately 12 Mt of mineralised material had been mined and by 1982 the annual production was approximately 1,270 t of tin metal per annum, along with 32 t of tantalite contained within the smelter slags (which graded at 7-9 % Ta_2O_5).

Historical production figures indicate that the mine produced a total of 37,000 t of tin and 3,000 t of tantalite from 27 Mt of mineralised material. Production figures for the period 1984-1992 are summarised in Table 1-2.

Table 1-2
Summary of tin production from 1984 to1992

Year	Tonnes milled per annum	Feed/head grade (% Sn)
1984-88	(Ave) 962,063	0.146
1989	897,547	0.119
1990	985,489	0.116
1991	1,000,740	0.108
1992	1,156,810	0.089

Source: *Begg (2008)*

1.4 Geology and Mineralisation

The historic Kamativi tailings dump is a man-made deposit that was generated as a site for the containment of tailings produced during the processing of tin mineralisation at the Kamativi Tin Mine, most recently owned by the ZMDC.

The Kamativi tailings were deposited over the period 1936 to 1994 and are derived from the mining and processing of the mineralised tin-bearing (spodumene-bearing **l**ithium-**c**aesium-**t**antalum ("LCT")) pegmatites.

At Kamativi, spodumene is the main lithium mineral present with lesser amounts of cookeite, zinnwaldite, petalite and amblygonite. Historical estimates of the size and lithium content of the tailings dump indicate that there is lithium mineralisation of potential economic interest.

The pegmatites at Kamativi form part of a larger regional pegmatite belt within the Dete-Kamativi Inlier and are hosted in the supracrustal gneisses and schists of the Kamativi Schist Belt that form part of the Palaeoproterozoic Magondi Belt. The Dete-Kamativi inlier is the western extension of the Magondi Belt exposed through younger Phanerozoic cover. The inlier comprises granodioritic orthogneisses, granites and highly deformed and metamorphosed supracrustal sequences that are divided in four northeast-southwest tectonostratigraphic belts. The four belts are known as the Kamativi, Tshontanda, Inyantue and Malaputese and overlie and are surrounded by the orthogneisses.

The pegmatites that were mined at the Kamativi Tin Mine are unzoned bodies ranging from <1 m to >30 m wide and frequently bifurcate and pinch out along strike.

The pegmatites are albitised and contain tin (in the form of cassiterite), lithium (in the form of spodumene and a variety of other lithium minerals as listed above) and tantalum/niobium (tantalite-columbite minerals) mineralisation. The tin mineralisation occurs as cassiterite patches within the thinner pegmatite dykes and in the thicker flatter pegmatites it is more evenly disseminated with higher grade patches within individual pegmatites. The thicker (>4 m) pegmatites contain the lithium minerals, mainly spodumene, which constitute up to 15 % of the pegmatite composition. Average grades range from 0.65-0.75 % Li_2O and the spodumene contains 7 % Li_2O and <0.5 % Fe.

Wolframite is also reported to occur in small amounts in thin veins and stockworks of milky-grey quartz with minor tourmaline.

1.5 Current Exploration

To date the exploration and test work on the Kamativi Project has been limited to the following:

- a review of the historical data for the Project (Project history presented in section 6);
- a topographical survey of the tailings dump;
- limited sampling on the top of the tailings dump:
    - three sampling campaigns were conducted by Lintmar on the Kamativi tailings. The first sampling campaign was done in 2015, with follow-up sampling done in 2016 and 2017

which involved sampling of pits and trenches to a maximum depth of approximately 1.5 m;

- Mineral processing and metallurgical testing.
- drilling and sampling of the dump to inform a Mineral Resource estimate (see Section 13);
- XRD analysis of the samples from the drilling.

1.6 Metallurgy and Processing

Two sampling campaigns have been conducted on the tailings dump at the Kamativi Project. During 2015 samples were taken from historical sampling pits and trenches. These samples together with a second batch of samples taken from across the dump during June 2016 and July 2016 were submitted for mineralogical and metallurgical test work.

Elemental, grade by size, particle size distribution, and mineralogical analyses were carried out by SGS Laboratories in Johannesburg.

The preliminary test work indicates that:

- the original head samples contain an average of 0.74 % Li_2O, 316 ppm Sn, 24 ppm Ta and 0.83 % Fe. The individual sample Li_2O, iron and tantalum grades are fairly consistent.
- mineralogical analysis indicates that the head material contains in the order of 6.4 % spodumene, which accounts for 71 % of the total Li_2O content. A concentrate containing 96.3 % spodumene (6.68 % Li_2O) can be produced by means of heavy liquid separation ("HLS") at an SG of 2.96. Of the feed material, approximately 5.6 % reports to the HLS sinks fraction. The spodumene within the sinks fraction is well liberated, with 78.2 % of the crystals being entirely liberated.

The mineralisation and distribution of the elements of interest has been adequately examined as well as some demonstration made as to the variability of these characteristics across the dump. It is noted that, with the exception of tin, the distribution of other elements is relatively uniform.

The concentrate produced at this SG, under ideal conditions is characterised by the following calculated parameters:

- Li_2O grade of 6.11 % at a recovery of 74.85 %
- Sn grade of 1144 ppm at a recovery of 38.74 %
- Fe grade of 0.93 % at a recovery of 20 %
- a total mass yield or recovery of 10.22 %.

Based on data associated with the magnetic separation scoping test; 73.44% of the iron was removed from the heavy liquid separation concentrate, with a corresponding Li_2O loss of 2.09%. A final concentrate assay was recorded as 7.08% Li_2O and 0.26% Fe.

Confirmatory test work indicated that the deportment of lithium is almost exclusively to spodumene and this lends the project to the improved extraction of lithium bearing minerals from the Kamativi resource. This due to the higher density of the mineral relative to bulk species such as quartz and plagioclase, allowing for beneficiation by means of dense media separation. An additional consideration is spodumene's amenability to flotation processes. The lithium content of spodumene is also significant relative to the majority of other lithium bearing minerals.

Confirmatory mineralogical test work has confirmed that spodumene in the main lithium bearing mineral

1.7 Adjacent Properties

The Property is located within a larger licence, ML No. 12 that includes the old Kamativi Tin Mine which is held by the ZMDC.

The rights to the hard rock lithium hosted pegmatite mineralisation within ML No. 12 are currently held in a JV between ZMDC and China Beijing Pinchang.

1.8 Mineral Resource Estimate

The Mineral Resource estimate was based on geochemical analyses, which were subjected to a quality assurance and quality control ("QA/QC") programme, and density measurements collected from pitting. A total of 116 vertical holes were drilled; one hole was not assayed because it went into rock beneath tailings in the first run. Therefore, 115 drill holes were used to estimate the Mineral Resource. The holes were drilled on a 100 m grid, with some areas to the southwest being drilled at closer spacing.

Two drilling methods were employed; namely a core drilling method and an auger drilling method. A total of seven holes were twinned with the two methods. Analyses of the twin hole data demonstrated that the data collected using the two methods do not show significant bias with respect to one another.

A three-dimensional wireframe of the dump was created by merging two wireframe surfaces representing the top and bottom of the dump. The top of the dump surface is based on surveyed points on the surface of the dump and around the boundary of the dump, as well as drillhole collar surveys.

The SnO_2, Ta_2O_5, Fe_2O_3 and Nb_2O_5 grades were estimated into a three-dimensional block model using ordinary kriging. The grades were estimated as a single domain, using 3 m composites in a horizontal plane, since the tailings were deposited and settled horizontally. The estimated grades validated well against the composite grades. An average density was applied to derive the tonnage.

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Resource is classified into the Indicated and Inferred categories as shown in Table 1-3.

The Mineral Resource is reported for the total tailings dump. The lowest Li_2O block estimate is 0.22 % which MSA considers has reasonable prospects for eventual economic extraction given the anticipated low-cost bulk mining and non-selective nature of tailings storage facility reclamation, and the outcome of initial metallurgical test.

It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and the use of economic parameters used to assess the potential for eventual economic extraction is not an attempt to estimate Mineral Reserves, the level of study so far carried out being insufficient with which to do so.

Table 1-3
Kamativi Lithium Tailings Mineral Resource for the Total Tailings Dump, 10 September 2018

Category	Tonnes	Density	Li_2O	SnO_2	Ta_2O_5	Fe_2O_3	Nb_2O_5
	(Millions)	t/m^3	(%)	ppm	ppm	(%)	ppm
Indicated	26.32	1.67	0.58	493	41	1.22	65
Inferred	0.30	1.67	0.62	544	45	1.45	62

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources which are not Mineral Reserves have no demonstrated economic viability.*

1.9 Conclusions and Recommendations

Mineralogical test work has confirmed that spodumene is the primary lithium bearing mineral. Preliminary metallurgical test work suggests the spodumene is recoverable through a combination of gravity and flotation methods and that a potentially saleable spodumene concentrate can be produced However more detailed work is required in order to establish the most efficient work flow and also a suitable method for the removal the iron from the concentrate.

The results of work completed on the Project to date warrant further exploration. The recommendations to be considered for subsequent exploration activities for the next two years on the Project are detailed below and summarised in Section 26.

Further exploration work is required in order to advance the Project and the proposed programme is:

- Infill auger drilling and sampling programme over the high-grade portion of tailings dumps to improve the confidence in the Mineral Resource.
- advanced metallurgical test work taking into account possible variations in grade and mineralogy identified in the drilling programme and optimise the gravity process, flotation, grind size, and removal of the iron from the final concentrate; and
- finalisation of the location of the new tailings disposal facility and proposed process plant. This will need to be done in conjunction with the Environmental Impact Assessment (“EIA”).

Table 1-4
Summary of proposed exploration programme for next phase of exploration

Items	Key Quantities	Budget (USD)	Proposed Deliverables
Auger drilling programme	Approximately 525 m drilling and assays, Mineral Resource estimate and geological consulting	55,000	20 drill holes averaging 25m depth. Mineral Resource estimate brought to code-compliant category
PEA		80,000	
Advanced metallurgical test work	Flotation, grind size determination, magnetic separation and gravity test work	150,000	Metallurgical process work flow
Environmental Impact Assessment	Hydrological study, hydrogeology, soils, water, biological field studies, tailings and pilot plant processing permitting.	100,000	Initiation of EIA Process

2 INTRODUCTION

2.1 Scope of Work

The MSA Group (Pty) Ltd ("MSA") has been commissioned by Zimbabwe Lithium Company (Mauritius) ("ZIM") to provide an updated Independent Technical Report on the Company's exploration activity on the Kamativi Lithium Tailings Property ("the Property") located in Matabeleland North, Zimbabwe. Chimata Gold Corporation ("Chimata") has entered into an option agreement with the ZIM. ZIM, through its wholly owned subsidiary Jimbata (Pvt) Limited, holds, or has the right to, a 60 % interest or other indirect interest in the Kamativi Tailings Company (Pvt) Limited through Joint Venture Agreement(s) with the Zimbabwe Mining Development Corporation ("ZMDC"), owners of Kamativi Tin Mines who have a 40 % interest in the Kamativi Tailings Company (Pvt) Limited.

The Property comprises an area contained within the Kamativi Mining Lease No. 12 ("ML No. 12") which is held by the ZMDC. Kamativi Tailings Company (Pvt) Limited have a 100 % interest in the Kamativi Lithium Tailings Project.

The main focus of exploration on the Property and the subject of this Independent Technical Report ("Report") is the Kamativi Lithium Tailings Project ("Kamativi tailings" / "Kamativi Project" / "the Project") which comprises the tailings dump derived from the historical tin mining conducted within ML No. 12 at the now disused Kamativi Tin Mine.

ZIM contracted The MSA Group (Pty) Ltd to execute an exploration drilling programme to inform a Mineral Resource estimate focussed on the tailings dump.

This report is to be utilised to enter into a Definitive Agreement with ZIM to eventually acquire 100 % of ZIM's interest in the Project. Chimata will be responsible for making the necessary funds available for the purpose of exploration and evaluation of the Property.

The report has been prepared in accordance with the disclosure and reporting requirements set forth in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP, Form 43-101F1, and the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on May 10, 2014.

A final draft of the report was also provided to ZIM, along with a written request to identify any material errors or omissions prior to lodgement.

ZIM's mineral Property is considered to represent an "Exploration Project" which is inherently speculative in nature. However, MSA considers that the Property has been acquired on the basis of sound technical merit. The Property is also generally considered to be sufficiently prospective, subject to varying degrees of exploration risk, to warrant further exploration and assessment of their economic potential, consistent with the proposed programmes.

Exploration and evaluation programme costs for the next phase of exploration are summarised in Section 26. Chimata will aim to raise sufficient working capital to ensure at least two years of operation. The funds raised in the initial placement are understood by MSA to be committed to the exploration and development of the Property in Zimbabwe.

ZIM has prepared staged exploration and evaluation programmes, specific to the potential of the Project, which are consistent with the budget allocations. The Project has evolved on the basis of a recently completed exploration drilling campaign over the last five months which forms the basis of this update and MSA considers that the relevant areas have sufficient technical merit to justify the proposed programmes and associated expenditure. The proposed Year 1 and 2 exploration budgets exceed the minimum annual statutory expenditure commitment on the mineral concessions in Zimbabwe.

All monetary figures expressed in this report are in United States of America dollars (“US$/USD”) unless otherwise stated.

Coordinates shown on maps and sections are relative to UTM35K/ARC1950 or UTM35K/WGS84 and metres above mean sea level (”mamsl”) unless otherwise stated.

2.2 Principal Sources of Information

MSA has based its review of the Property on information provided by ZIM, along with technical reports by Government agencies and previous tenement holders, a report titled “NI 43-101 Technical Report For the Kamativi Lithium Tailings Project, Kamativi, Matabeleland North Province, Zimbabwe” prepared by Cronimet Mining Processing SA (Pty) Ltd, a sub-contractor commissioned by ZIM to undertake preliminary technical feasibility work, and other relevant published and unpublished data. Data generated during the exploration drilling campaign is 2018 forms the basis for the update to this Independent Technical Report. A listing of the principal sources of information is included at the end of this Independent Technical Report (Section 27).

2.3 Qualifications, Experience and Independence

MSA is an exploration and mining consulting and contracting firm, which has been providing services and advice to the international minerals industry and financial institutions since 1983.

This Technical Report has been compiled by:

- Mr Michael Cronwright (B.Sc. Hons., M.Sc.; FGSSA; Pr.Sci.Nat.), who is a professional geologist with 19 years’ experience, the majority of which has involved the regional mapping and exploration on a wide range of commodities, primarily within southern Africa, including Competent Person (“CP”) oversight, reviews, Qualified Person (“QP”) and due diligence studies of tin properties in East Africa, numerous lithium projects in southern Africa and detailed mapping and a review of the Alto Ligonha Pegmatite Province in northern Mozambique. He is a Principal Consultant for MSA, is registered as a Pr.Sci.Nat. with the South African Council for Natural Scientific Professions (“SACNASP”) and a member in good standing with SACNASP and is a Fellow of the Geological Society of South Africa (“GSSA”). Mr Cronwright has the appropriate relevant qualifications, experience, competence and independence to act as a “Qualified Person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects). Mr Cronwright is responsible or co-responsible for Sections 1-13, 19 and 24-27; and

- The Mineral Resource estimate has been completed by Mrs Ipelo Gasela (Pr. Sci Nat.) who is a geologist with 13 years' experience in Mineral Resource evaluation and reporting. Mrs Gasela (BSc Hons, MSc (Eng.)) is a Senior Mineral Resource Consultant for The MSA Group (an independent consulting company), is registered with the South African Council for Natural Scientific Professions (SACNASP) and is a Member of the Geological Society of South Africa (GSSA). Mrs Gasela has the appropriate relevant qualifications and experience to be considered a "Qualified Person" for the style and type of mineralisation and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects. Mrs Gasela is responsible or co-responsible for sections 1, 14; and 25-27.
- Mr John Derbyshire (B.Sc. Eng. (Chem), Pr.Eng., FSAIMM) is a Professional Engineer with more than 37 years' experience in plant operations and projects over a range of commodities. Mr Derbyshire has worked as a Metallurgical Consultant and Manager for many years with responsibilities including program co-ordination, monitoring and interpretation of metallurgical output, and operational responsibilities including metallurgical aspects of design, construction and commissioning and operation of concentrators. He is an Associate Metallurgical Consultant for MSA, is registered as a Professional Engineer with the Engineering Council of South Africa and is a Fellow of the South African Institute of Mining and Metallurgy. Mr Derbyshire has the appropriate relevant qualifications, experience, competence and independence to act as a "Qualified Person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects). Mr Derbyshire is responsible or co-responsible for of sections 1, 13, 17 and 25-27.

Peer review has been undertaken by Mr Mike Robertson and Mr André van der Merwe. Mr Robertson is a professional geologist with 27 years' experience on base metal and gold exploration projects throughout Africa, as well as other regions globally. His experience covers all aspects of exploration project execution and management, with an emphasis on compliance with minerals industry reporting codes. He also has considerable consulting and public reporting experience. Mr Robertson is registered as a Pr.Sci.Nat. with SACNASP and is a Fellow of the GSSA. He is a Principal Consultant at MSA and is based in the Johannesburg office.

Mr André van der Merwe is a professional geophysicist and geologist with almost 30 years' experience in exploration, mining, project development (feasibility studies), due diligence reviews and valuations of mineral assets. Mr van der Merwe has been Technical Advisor to several successful listings on FTSE, AIM, TSX, ASX and JSE, as well as private fundraisings. He has provided Independent Expert Reports for several ASX listed companies. Mr van der Merwe, Head of Mining Studies at MSA, is registered as a Pr.Sci.Nat. with SACNASP, a Member of the Australasian Institute of Mining and Metallurgy and is a Fellow of the GSSA.

Neither MSA, nor the authors of this report, have or have had previously, any material interest in Chimata or the mineral properties in which Chimata has an interest. The QPs are not insiders, associates or affiliates of BLE. MSA's relationship with Chimata is solely one of professional association between Client and Independent Consultant. This Report is prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this Report.

2.4 Site Visits

A site visit was made by Mr Michael Cronwright during the period 7 to 8 August 2017 to the Property during which check samples were taken on the tailings dump previously sampled by ZIM's local representative, Lintmar. A second site visit was made by Mr Michael Cronwright during the period 23-24 April 2018. MSA has endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the Independent Technical Report is based.

2.5 Effective Date

The Independent Technical Report has been prepared on information available up to and including 10 September 2018.

3 RELIANCE ON OTHER EXPERTS

MSA's opinion contained herein is based on information and data provided to MSA by ZIM and their partners throughout the course of the investigations. MSA have relied on ZIM for input on Property ownership, history, geology, exploration, permitting and market studies in support of this Technical Report.

MSA used their experience to determine if the information from previous reports was suitable for inclusion in this Report and adjusted information that required amending. This report includes technical information which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, MSA do not consider them to be material.

MSA has relied on ZIM for information relating to Property ownership and agreements. The Project is understood to consist of an area, of approximately 5.9 km^2 within ML No. 12, which covers an area of approximately 44 km^2, issued to Kamativi Tin Mines (a subsidiary of ZMDC) on the 5th of January 1976.

The preparation of Section 13 of this report has relied on reports generated by various accredited external laboratories and on reporting of the magnetic separation work performed by the in-house metallurgical staff of the clients engineering company CMPSA, in conjunction with Multotec. The outcomes of the latter are contained in a document entitled "2018 11 02 Kamativi Technical Facts and Figures Memo" generated by CMPSA and relate to the separation of impurity Fe from the HLS concentrates. Further work in this regard is to be carried out during the ongoing project.

MSA has not independently verified, nor is it qualified to verify, the legal status of this licence and/or any other licences which have been amalgamated into the current licence. The present status of the licence(s) listed in this report is based on information and copies of documents provided by ZIM and its partners, and the report has been prepared on the assumption that the tenements will prove lawfully accessible for evaluation. MSA did not seek an independent legal opinion on these items.

Neither MSA nor the authors of this report are qualified to provide extensive comment on legal issues associated with Chimata's joint venture agreements. Comment on these agreements is for introduction only and should not be relied on by the reader.

No warranty or guarantee, be it express or implied, is made by the QPs with respect to the completeness or accuracy of the legal aspects of this document. The QPs do not undertake or accept any responsibility or liability in any way whatsoever to any person or entity in respect of this part of this document, or any errors in or omissions from it, whether arising from negligence or any other basis in law whatsoever.

Neither MSA nor the authors of this report are qualified to provide comment on environmental issues associated with ZIMs Property and the Project.

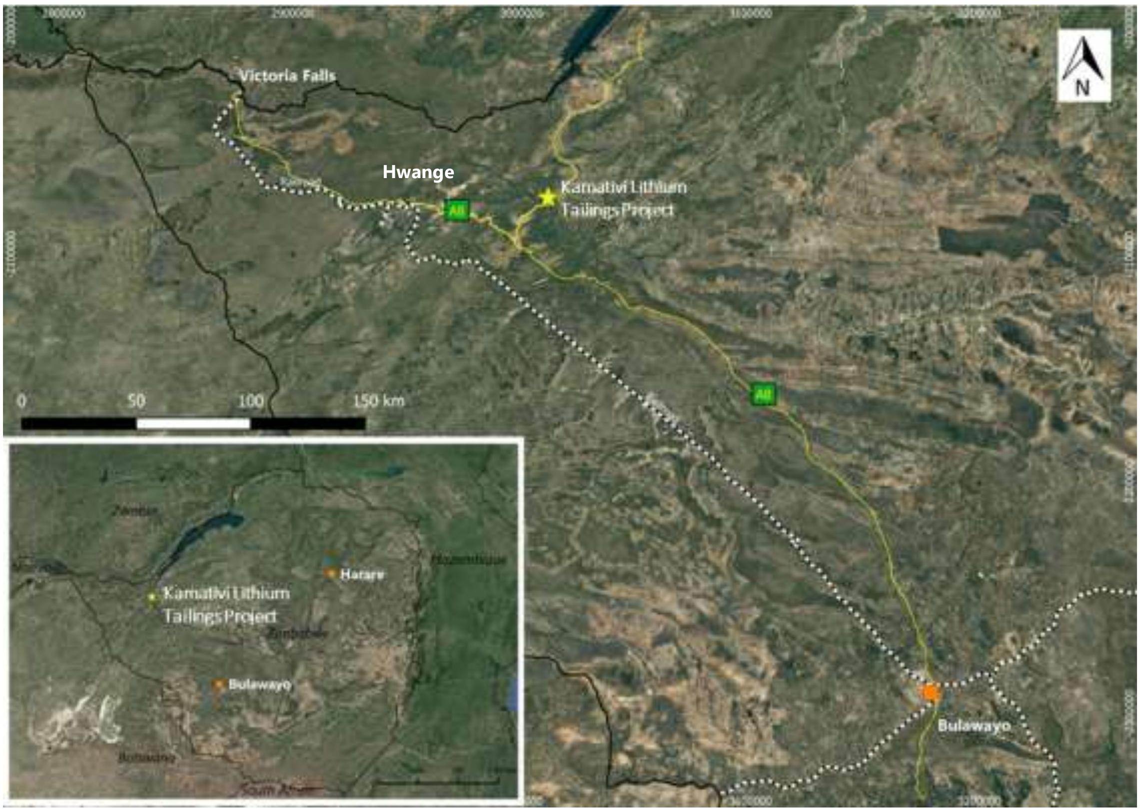
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Kamativi Lithium Tailings Project (the "Project") is located outside the village of Kamativi in Matabeleland North Province, Zimbabwe. The Project, which is identified as a tailings deposit associated with the disused Kamativi tin mine, is located approximately 185 kilometres east-south-east of Victoria Falls, approximately 84 km by all-weather tar road east of Hwange and approximately 310 km northwest of Bulawayo.

The Project is situated within the Kamativi Tin Mining Lease No. 12. Access to the mine from Bulawayo is along the A8, tarred Bulawayo/Victoria Falls highway up to the 270 km peg, from which a north bound tarred road is followed for 29 km to the mine. The Project location is shown in Figure 4-1.

Figure 4-1
Map showing the location of the Property within Zimbabwe



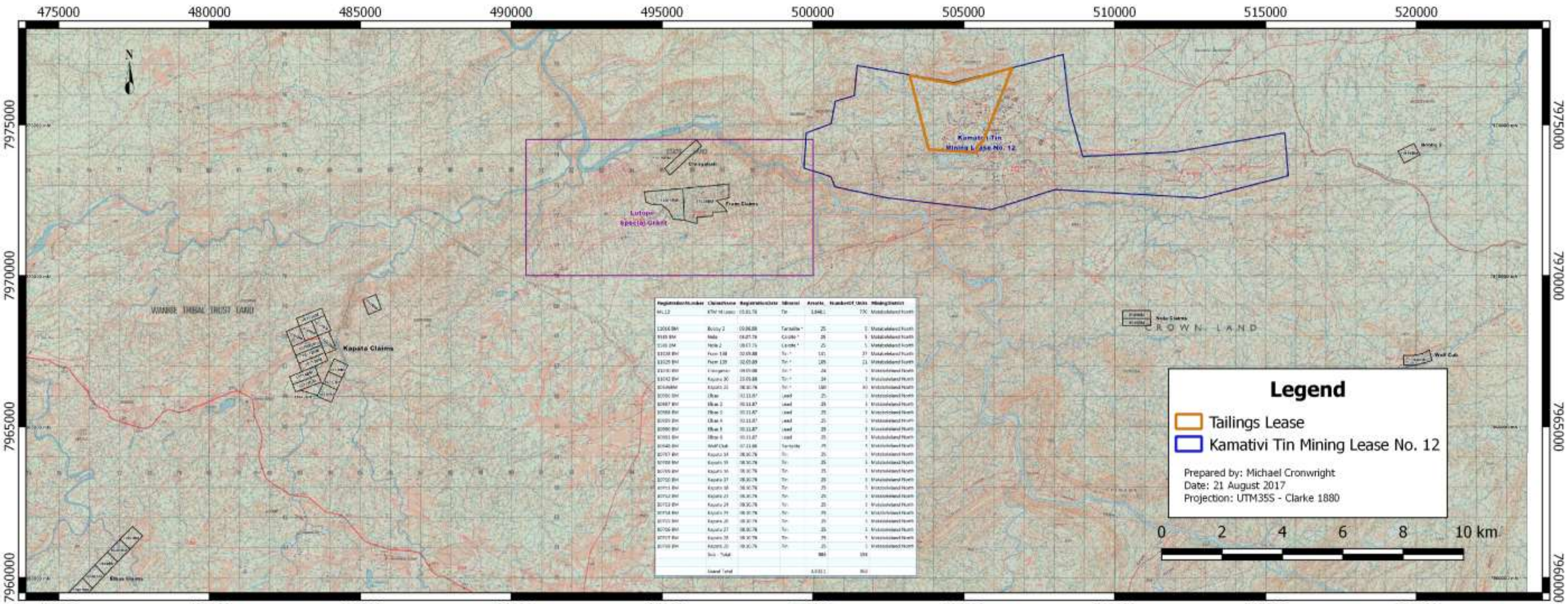
Note: *Background map sourced from Google Earth (2017)*

4.2 Mineral Tenure, Permitting, Rights and Agreements

The Project comprises an area of 5.91 km^2 covering the tailings dump from the old Kamativi Tin Mine and falls within ML No. 12 which covers an area of 44.4 km^2 (Figure 4-2 and Figure 4-3).

Figure 4-2

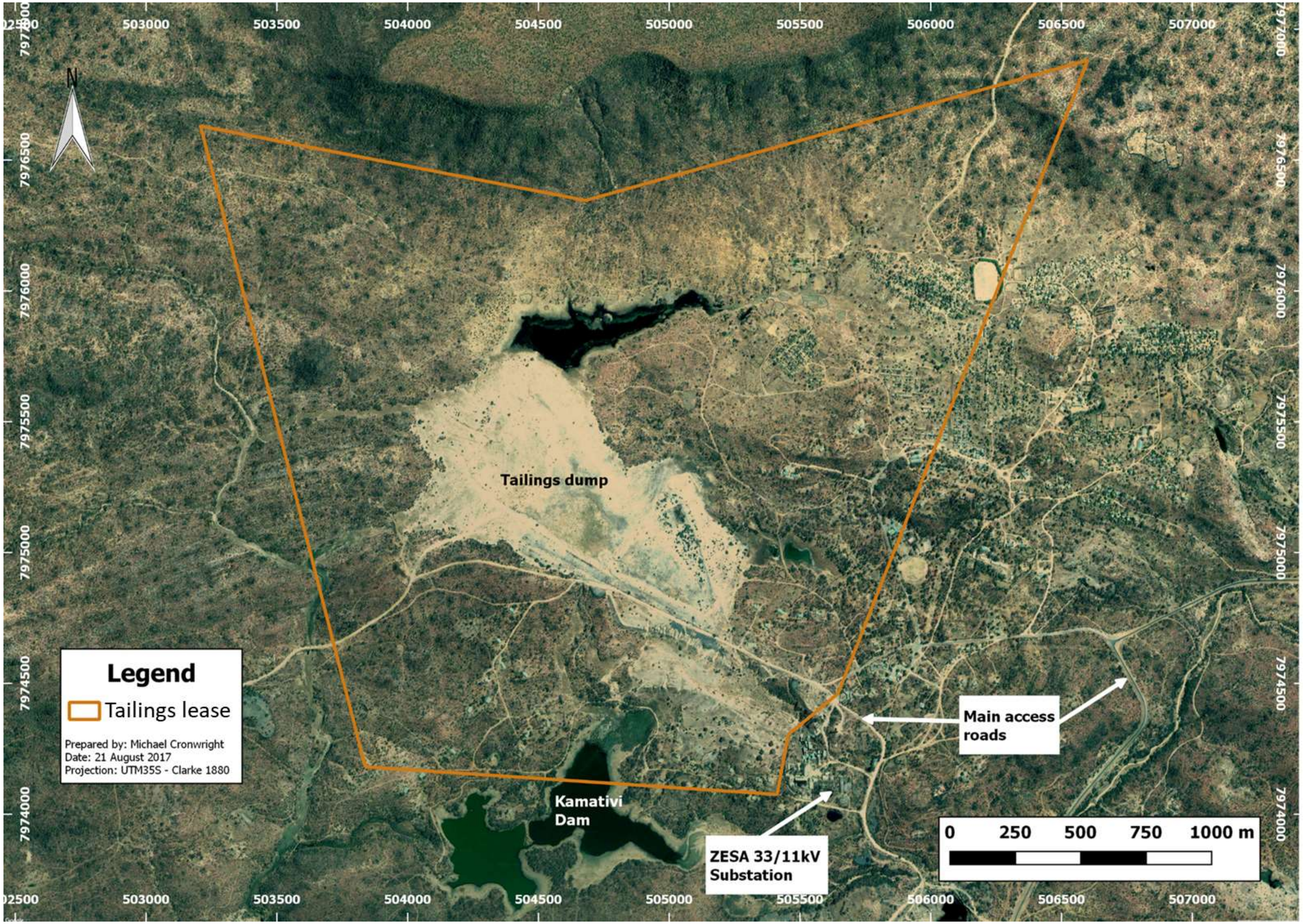
Map showing the location of the Property within ML No.12. Also shown are neighbouring properties in the area



Source: *ZMDC (2014). Background composite map (Zimbabwe 1:50 000 scale topographic maps: 1826 B3 Wankie East, 1826 B4 Cross Roads, and 1827 A3 Kamativi)*

Figure 4-3

Map showing the outline of the Property and local infrastructure within ML No.12.



Note: *Background image sourced from Google Earth (2017)*

The mining lease was granted to Kamativi Tin Mines on the 5th of January 1976 for all minerals. The Zimbabwe Mining Development Corporation ("ZMDC") was established by the Act of Parliament No. 31 of 1982 and is owned by the Government of Zimbabwe. On November 24, 1986, the Industrial Development Corporation of Zimbabwe ("IDC") transferred 91.3 % the shares it owned in Kamativi Tin Mines Limited to the ZMDC.

The Kamativi project is a joint venture ("JV") between the ZMDC, owners of Kamativi Tin Mines, which holds 40 % of the Project, and Jimbata (Pvt) Ltd ("Jimbata"), a company incorporated under the laws Zimbabwe, which holds 60 %. A JV agreement was entered into between Lintmar (Private) Limited ("Lintmar"), a company incorporated under the laws Zimbabwe, and ZMDC on February 2, 2018 (the "JV Agreement") (Appendix 2).

On February 6, 2018 Lintmar requested for ZMDC's approval in order to assign all of its rights and interests in the JV Agreement, as this assignment option was provided in said JV Agreement (Appendix 3). On February 14, 2018, following the aforementioned request by Lintmar, ZMDC confirmed its consent to the cession, assignment and transfer by Lintmar to Jimbata of Lintmar's rights, obligations and interest of the JV Agreement (Appendix 4). By a letter dated February 16, 2018, Jimbata confirmed the cession by Lintmar of all of its rights and interests in the Kamativi Mine Tailings Dump to Jimbata, including any and all ancillary rights in the JV Agreement (Appendix 5). Jimbata is 100 % owned by the Zimbabwe Lithium Company (Mauritius) Limited ("ZIM"), a privately held company incorporated under the laws of Mauritius.

The JV company (being respectively owned by Jimbata (60 %) and Kamativi Tin Mines (40 %)), named Kamativi Tailings Company (Pvt) Limited, was incorporated on February 16, 2018 as per the *Companies Act [Chapter 24:03] of Zimbabwe*.

On February 14, 2018, Chimata announced by way of press release (a copy of which is reproduced as Appendix 10) that it had entered into a binding letter of intent ZIM (the "LOI"). In terms of the LLOI, Chimata will subscribe by way of share exchange for an initial subscription of 19% of ZIM's share capital in exchange for the allocation by Chimata of an amount of shares representing 19 % of its then outstanding share capital to ZIM. Chimata has the right to further acquire the remaining issued and outstanding share capital of ZIM upon fulfilling of certain terms and conditions as set out in the LOI, the whole resulting in ZIM becoming a wholly owned subsidiary of Chimata.

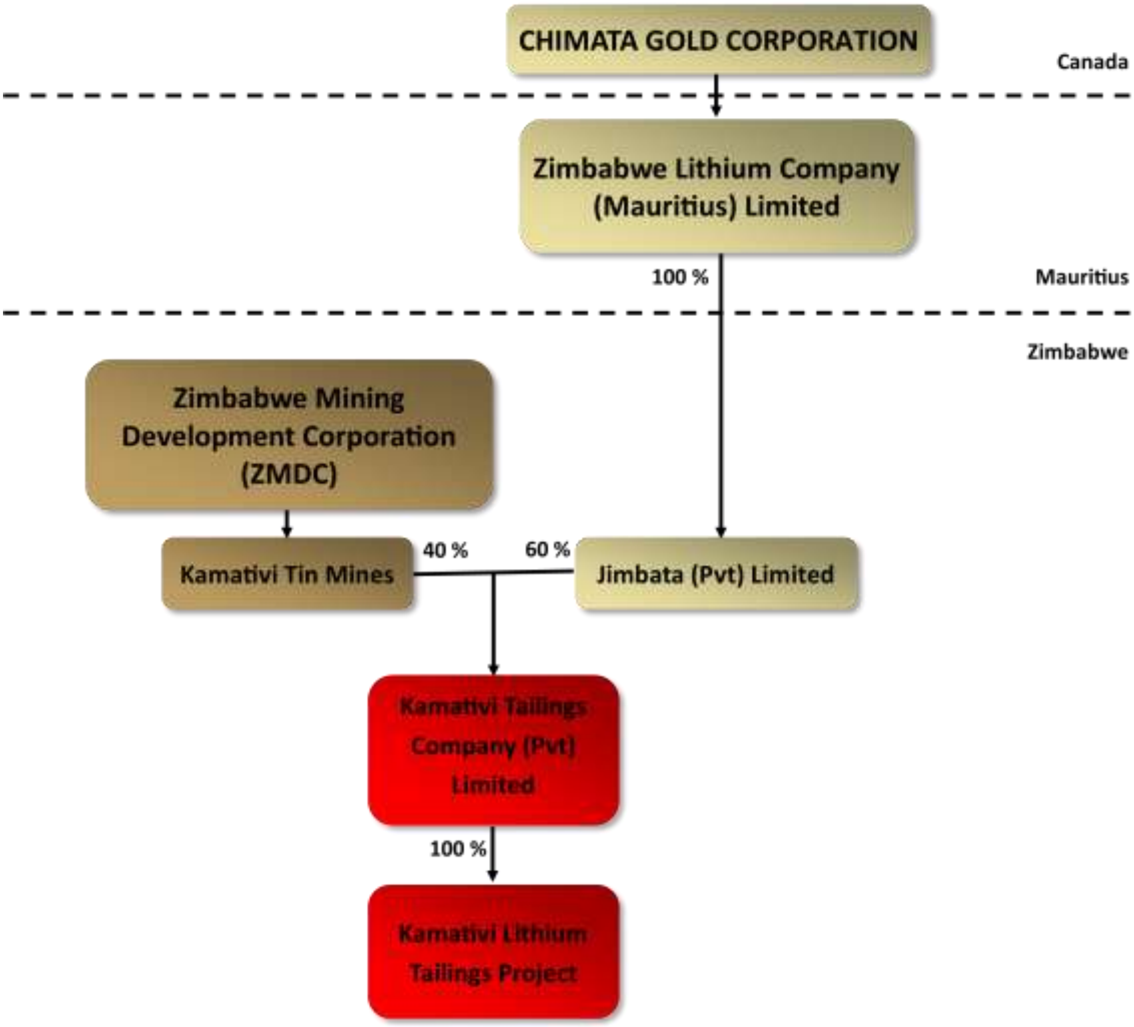
On positive outcome of this NI 43-101 Technical Report, Chimata and ZIM will enter into a definitive share exchange agreement which will give Chimata the right, on completion of a Mineral Resource estimate undertaken in accordance with NI 43-101, and the fulfilment of the requirements as set out in the LOI, to acquire 100% equity ownership of ZIM. This would give Chimata ownership over ZIM's equity interests, ZIM being a 100 % equity owner of Jimbata, as mentioned hereinabove.

In terms of the JV Agreement, Jimbata are due to pay ZMDC the first option payment on signing of the JV agreement and ZMDC's transfer of the mineral title to Kamativi Tailings Company (Pvt) Limited. A second option payment will be paid six (6) months after the signing of the JV Agreement and then a third payment nine (9) months after signing the JV agreement.

The JV Agreement between ZMDC and Jimbata allows for Kamativi Tailings Company (Pvt) Limited to undertake a full Feasibility Study on the tailings material and, should the Feasibility Study have a positive outcome, allows for the construction of a concentrator plant to beneficiate the tailings material, producing a lithium concentrate and any other viable by-products that may be able to be produced. This JV Agreement is for the life of dump and is anticipated to be completed over a 10-year period.

The ownership structure of the Project is shown in Figure 4-4.

Figure 4-4
Project ownership structure



4.3 Environmental Liabilities

MSA is not aware of any special environmental restrictions or liabilities related to the Project. In terms of the JV agreement, Jimbata will not be liable for any historical environmental liabilities.

4.4 Major Risks

According to the 2017 RiskMap published by Control Risks, Zimbabwe is classified as having a high political risk factor and a medium security risk factor.

Kamativi Tailings Company (Pvt) Limited will be applying for Special Economic Zone status which will provide various investor and ownership protection instruments. Special Economic Zones Act (Chapter 14-34) is an act to provide for the establishment of the Zimbabwe Special Economic Zones Authority whose functions will be to provide for the establishment of special economic zones, the administration, control, regulatory measures and incentives in connection there with. The proposed incentives for investors include exemption on certain taxes, royalties and dividends and tax holidays on capital equipment. Every licensed investor carrying on an approved activity within a special economic zone may, subject to any approval required under the Exchange Control Act (Chapter 22-05) operate a foreign currency account with any banking institution. In addition, the Indigenous and an Economic Empowerment Act (Chapter 14-33) shall not apply in relation to licensed investors operating in a special economic zone

It is the intention of Jimbata to obtain licensed investor status and have the Project designated as a special economic zone thereby providing the protection regarding various risk factors.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Introduction

An understanding of the local climate, physiography and existing infrastructure and their effects on the socio-economic status of the community and the processing operation as a whole is essential to the future success of the Project. Information presented in this section of the document is sourced from the literature and data gathered from an initial site visit completed by MS Montan Services UG and Green Resources Company (Pvt) Ltd. towards the end of August 2016.

5.2 Accessibility

The Kamativi Lithium Tailings Project is located outside the village of Kamativi in Matabeleland North Province, Zimbabwe. Access to the Project area from the town of Hwange is via an approximately 84 km all-weather tar road, travelling along the tarred A8 in an east-southeast direction from Hwange before turning onto the Kamativi road at the A8-Dete-Kamativi crossroads and travelling northeast for approximately 30 km to the Kamativi mine.

The proposed route from Hwange is shown in Figure 5-1 and the slightly longer route from Victoria Falls to Kamativi is shown in Figure 4-1. The village of Kamativi is accessible via an unpaved road from the mine.

Figure 5-1

Locality map of the Kamativi Project showing main road and rail access routes

Note: *Background map sourced from Google Earth (2018)*

5.3 Climate and Physiography

The Project area is covered by a combination of grassland, scrub and woodland areas restricted to the river valley and hills (Figure 5-2). The climate in the region is classified as a hot semi-arid climate (BSh) according the Köppen and Geiger classification which is characterized by extremely variable temperature ranges and relatively little precipitation.

The average temperature for the year in Kamativi is 22.3 °C. The warmest month, on average, is October with an average temperature of 26.1 °C. The coolest month on average is June, with an average temperature of 16.6 °C.

Most of the rain falls between November and March and the average annual precipitation is 635 mm. The month with the most precipitation on average is January with 175 mm of precipitation. The months with the least precipitation on average are July and August with an average of 0 mm. There is an average of 69.3 days of precipitation, with the most precipitation occurring in January with 15.1 days and the driest months being June and July little to no precipitation falls.

The warmest month with an average temperature of 26.5 °C is October and June is the coldest month, with an average temperature is 16.4 °C (https://en.climate-data.org/location/772745/).

Figure 5-2

The typical topography and vegetation of the Property

The topography of the Property is undulating with prominent ridges and hills which comprise more competent and resistant rocks (Figure 5-2). The general drainage network is dominated by small, ephemeral rivers that flow only when it rains and larger non-perennial rivers like the Kamativi River which flows through the Property (east to west). The Kamativi flows into the Gwayi River to the southwest of the Property. The Kamativi River is dammed to form the Kamativi Dam, located to the south of the tailings dump, straddling the southern boundary of the Tailings licence area and the Gwayi River (Figure 4-3 and Figure 5-1).

A visit to the tailings deposit itself showed a sandy substrate with a grain size of estimated 0 – 1000 micron in size. As a result of 20 years of renaturation a meagre grass cover had developed on the sandy ground, together with some shrubs and small trees. The vegetation cover is shown in Figure 5-2 and Figure 5-3.

Figure 5-3

Vegetation on the Tailings Dam after 20 years of revegetation

5.4 Local Resources and Infrastructure

5.4.1 Amenities

The village of Kamativi is located within ML No. 12 and has a population of between 1,500 and 6,000 (http://travelingluck.com/Africa/Zimbabwe/Zimbabwe+(general)/ 889390 Kamativi.html), depending on the information source. The community is served by tuck shops which stock basic groceries. Prior to the closure of Kamativi Mine in 1994, ZMDC operated a large supermarket. The vast majority of supplies will have to be sourced from larger towns and/or cities within the region such as Hwange and Victoria Falls (Figure 4-1 and Figure 5-1), with more specialised items being procured from larger cities such as Bulawayo and Harare.

Residents of Kamativi receive medical treatment at the local Kamativi Mission Hospital which is administered by the Hwange Roman Catholic Diocese. The hospital is staffed by qualified medical personnel. In addition to primary care facilities, the hospital includes a maternity wing and a mortuary. The hospital has the capacity to look after the medical needs of a large community.

Historically, ZMDC owned 571 houses which it has now given to the Hwange Rural District Council to administer. Kamativi Tailings Company (Pvt) Limited will need to consult and negotiate with the Hwange Rural District Council regarding the availability of and right to use the housing. The housing comprises:

- 84 low-density units with three or four bedrooms;
- 24 medium-density units with two bedrooms;
- 463 high-density units; and
- 212 one-roomed single quarters.

The existing sewage plant is not operational and is in need of rehabilitation. As a result, it poses a health hazard to the high- and medium-density suburbs that used to rely on it. The low-density suburbs use septic tanks.

The Matabeleland North region supports dryland ranching and game-farming as principal agricultural activities. Due to the low rainfall and poor soil in the region, commercial crop farming is not possible and rural farmers often struggle to support their families.

The region is readily accessible via a well-maintained road, rail and air transportation network.

5.4.2 Transport infrastructure

The Project is located approximately 60 km east of Hwange (in a straight line) and accessed via the tarred A8 road for about 55 km in an east-southeast direction and then in a northeasterly direction for approximately 30 km along a secondary tarred road (Figure 5-1 and Figure 5-4).

Figure 5-4

Tarred road from the A8-Dete-Kamativi crossroads to Kamativi

The closest railhead, which is in need of repair, is located at the town of Dete (Figure 5-5), approximately 45 km southwest of the Project area, via a tarred road. The railhead is located on the Bulawayo-Hwange-Victoria Falls railway line which is operational and serves the Hwange coal fields. The Dete siding has a dedicated spur that previously served the district and the Kamativi mine.

Figure 5-5
Aerial image of the Dete railway siding

Source: *Image sourced from Google Earth – Image © 2018 CNES / Airbus (2018)*

The closest international airports to the Project are situated in Bulawayo (IATA Code - BUQ) and Victoria Falls (IATA Code - VFA) (Figure 5-6) and are serviced by South African Airways, British Airways, Air Zimbabwe and other regional airlines. Suitable hotels and lodge accommodation exist with easy access to the mine. Medical facilities are available in Hwange, Bulawayo and Victoria Falls. The Hwange National Park is also serviced by a domestic airport with a blacktop runway and is 50 km from the Project.

The domestic airport is 5 km away from the Hwange Safari Lodge, an upmarket tourist destination serviced by the Legacy Group of Hotels. Various other lodges in the area support tourism.

Figure 5-6
Photograph of the Victoria Falls International airport

5.4.3 Power Infrastructure

The old Kamativi mine and village is fed by a 33 kV overhead powerline from Hwange. A 33/11 kV substation (Figure 4-3 and Figure 5-7) is located near to the old Kamativi processing plant with a single 2,000 kVA 33/11 kV transformer installed (Figure 5-8). The transformer has a 2,200V tertiary winding which appears not to be connected. This substation is equipped to supply two transformers, one of which has been removed. Reticulation around the site includes five 11 kV overhead lines with separate transformers and meters.

Figure 5-7
Kamativi Mine substation

Figure 5-8
Kamativi Mine transformer

5.4.4 Water

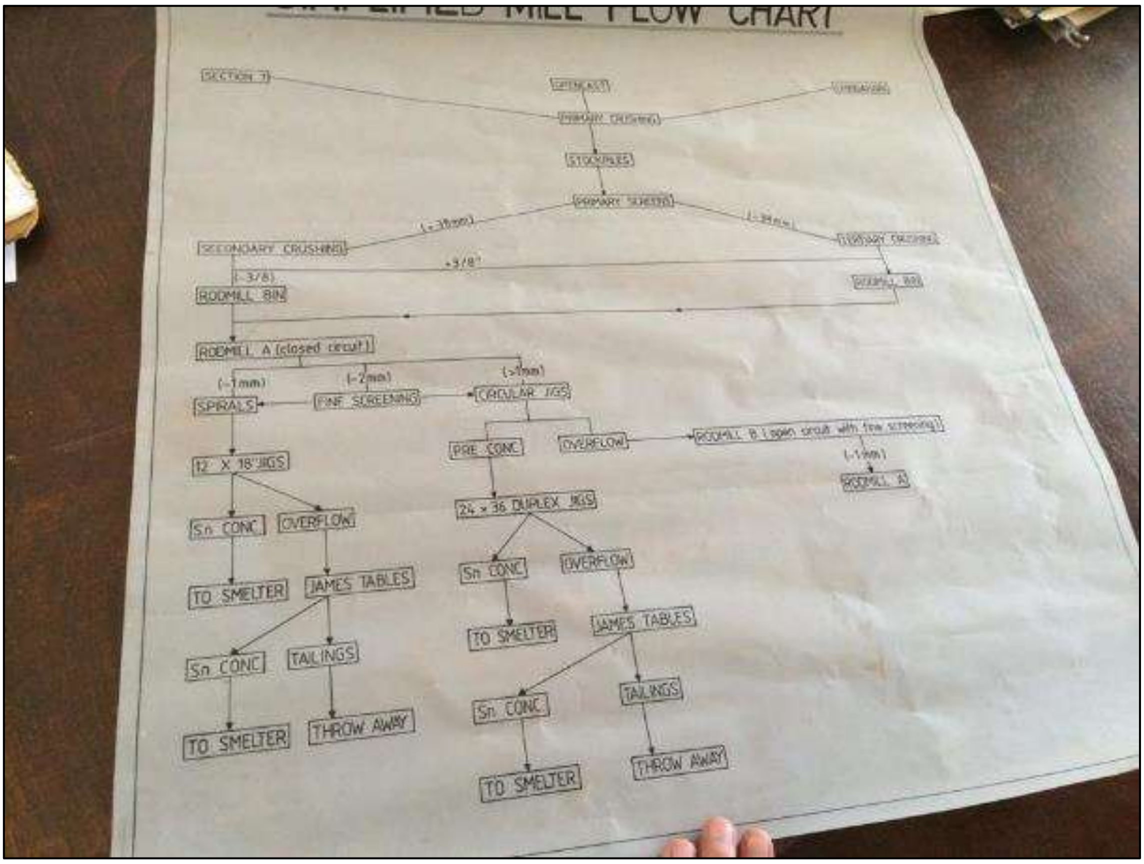
Water for household use is untreated and pumped from the Kamativi dam and three boreholes in the area, the boreholes each sourcing water from a depth of approximately 100 m. One borehole is exclusively for the supply of water to the clinic. The remaining two boreholes supply water to the majority of the population. Both these boreholes are known to be contaminated and residents are forced to boil or chemically treat the water. Due to the Hwange Rural District Council's lack of funds, the water cannot be treated and consequently the residents are prone to contracting waterborne diseases, commonly diarrhoea, dysentery and cholera. It was reported that no waterborne sewage system is in place. The sewage is reported to be led into the Kamativi Dam.

Historically, the Kamativi mining operation obtained its water for processing from the Kamativi dam (Figure 4-3) situated, south of the mining town and built on the Kamativi River. Water from the Kamativi dam was supplemented with water from the Gwayi River (to the southwest of the Property), sourced from a weir built midway between the old and new bridges that span that river. There is evidence in the field of the eye bolts and concrete foundations which used to secure the pumps and pipework. The 33 kV power line crosses the river at this point too and could be used (and probably was used when the mine was in operation) to supply power to the pump if this water is required for the proposed operations. A backup power supply should also be considered if necessary.

The processing for the historical tin extraction exclusively used gravity separation. The Flow Chart presented by the ZMDC representative at site, as shown in Figure 5-9, does not indicate any risk of chemical hazards. However local newspapers report a high level of chemical contamination of the water in a nearby dam; "One of the dams - once an open cast site - was reportedly condemned by the Ministry of Health and Child Care a few years ago because its water is contaminated with mining chemicals." This still needs to be independently verified.

Figure 5-9

Kamativi Tin Processing Flow Chart



Samples which had been taken earlier from the tailings dam indicated background naturally occurring radioactive materials ("NORM") including traces of Uranium (U) (average 12,0 ppm) and Thorium (Th) (average 2,28 ppm). Studies indicate that the NORM levels at Kamativi are lower than for any other pegmatite areas, not only in Zimbabwe but globally. This low radiation level will be verified in the Social and Environmental Impact Assessment ("SEIA") study. Monazite is only mineral containing any radioactive elements (i.e. Th and U) according to the Mindat (www.mindat.com) list of minerals found in the Kamativi mine. Monazite is considered a resistate, inert mineral and not water soluble, and is thus not considered a potential environmental hazard or source of the supposed water contamination.

5.4.5 Processing and Waste Disposal Sites

The Project covers an area of 613 hectares and should have sufficient space for processing and waste disposal sites but will need to be confirmed through future work.

5.5 Human Resources

The Kamativi Tin Mine was the main source of employment in the area and on its closure in 1994, most of the skilled personnel left the area. Currently the only employment at Kamativi is provided by the ZMDC, for staff responsible for the administration and the care and maintenance. It is expected that labour will be drawn from the existing village population, supplemented by skilled and management personnel from the Hwange region where there are a number of operational coal mines and processing facilities.

In addition, wildlife tourism (based around the Hwange National Park to the west of the Project, other smaller nature reserves and Victoria Falls to the north-west) is a significant source of employment in the region.

There is a primary school in the area, which was previously funded by ZMDC and is now funded by the local Parents Association, as well as the St Teresa Primary School, which is owned by the Roman Catholic Church. There is a single ZMDC funded secondary school in the area for approximately 300 pupils. It is a day school with many of its pupils coming from areas beyond Kamativi.

There are no active tertiary education facilities (colleges or universities) within the immediate vicinity of Kamativi. The closest it the newly built university at Lupane, approximately 180 km away, which has not yet commenced enrolment or studies. Bulawayo has the Zimbabwe School of Mines and the New University of Science and Technology ("NUST") which is normally a potential source of technical graduates.

6 HISTORY

6.1 Discovery, Historical Exploration Work and Mining History

The first reports of tin in the region date back to about 1920. The first claims were pegged in 1935. Production of cassiterite concentrates from alluvial deposits and surface rubble commenced in 1936. The history of mine ownership is summarised below and sourced from (Anon, (1963), Anon (1996) and http://www.portergeo.com.au/database/mineinfo.asp?mineid=mn1537):

- **1920–1935:** Reports of tin occurrence in region and first pegging's of claims by Mr R. H. Aldworth in 1935;
- **1936-1944:** First production declared by Mr R. H. Aldworth in partnership with Mr S. Sauerman who commenced hydro-sluicing and panning of the surface rubble and alluvial deposits;
- **1939-1949** (including the Second World War): Production increased when various small mills were erected and a smelter built in Bulawayo. After the war, tributors under the Government Ex-Serviceman Rehabilitation Scheme, worked the claims on selective underground mining;
- **1949-1951**: Rhodoak Limited set up the first Kamativi Tin Mines Limited with a capital investment of £ 200,000;
- **1951:** Oakes Trust acquired Kamativi Tin Mines Limited;
- **1952:** N.V. Billiton Maatschappij, a Dutch Naamloze Vennootschap ("N.V.") or public company, assumed financial and technical control and increased the authorised capital to £ 700,000 and later to £ 3,000,000. They installed a 400 tonne per day ("tpd") pilot plant, together with an oil-fired drum-type rotary smelter. By 1964 they were processing an average 42,000 tonnes of ore per month. The professional co-operation brought about contact with Billiton's Tin Smelter at Arnhem, Netherlands where their metallurgists and chemists in "modern laboratories" assisted in extending production of the smelter to other products, i.e. tin solder and bearing metals;
- **1970–1986:** Industrial Development Corporation of Zimbabwe ("IDC") acquired a controlling interest and ultimately the entire Billiton Shareholding. Mill capacity increased to 2,000 tpd, producing 950 tonnes of refined tin and alloys per year and approximately 23 tonnes of tantalite (Ta_2O_5) in slags. The plants processing capacity increased from 56,000 tonnes per month ("tpm") to 80,000 tpm during this period:
    - by 1977 approximately 12 Mt of mineralised material had been mined and by 1982 the annual production was approximately 1,270 t of tin metal per annum, along with 32 t of tantalite contained within the smelter slags (which graded at 7-9 % Ta_2O_5); and
- **1986-1994:** The IDC transferred its interest of 91.3 % in Kamativi Tin Mines Limited to Zimbabwe Mining Development Corporation ("ZMDC"). During this period, the economic reserves became depleted and tin production decreased to 1,060 t in 1991 and 800 t in 1993. In 1994 the mine closed down due to low tin prices, following the tin price crash of 1985, coupled with falling ore grades.

Historical production figures from Begg (2008) indicate that the mine produced a total of 37,000 t of tin and 3,000 t of tantalite from 27 Mt of mineralised material. Begg (2008) also lists the production figures for the period 1984-1992 as summarised in Table 6-1.

Table 6-1
Summary of tin production from 1984 to1992

Year	Tonnes milled per annum	Feed/head grade (% Sn)
1984-88	(Ave) 962,063	0.146
1989	897,547	0.119
1990	985,489	0.116
1991	1,000,740	0.108
1992	1,156,810	0.089

Source: *Begg (2008)*

Historical Reserve estimates quoted by Begg (2008) could not be verified. The most recent historical estimate was prepared in 1994 and is summarised in Table 6-2.

Table 6-2
Summary of the 1994 historical estimate for underground and opencast workings

	Category	Tonnes	Sn grade (%)
Underground	Proven	1,915,000	0.189
	Probable	6,194,000	0.168
	Possible	15,202,000	0.177
	Reclamation	397,000	0.198
	Total	**23,708,000**	**0.176**
Opencast	Probable	2,300,000	0.189
	Possible	2,135,000	0.202
	Total	**4,435,000**	**0.195**
GRAND TOTAL		**28,143,000**	**0.179**

Source: *Begg (2008)*

6.2 Historical Mining and Processing Methods

Most of the plant feed (approximately 70 %) was supplied by the opencast workings. Waste stripping was done by bulldozers and supplemented with blasting when required. Broken mineralised material was loaded by mechanical shovel and transported by truck trucks to the mill and gravity processing plant (Anon., 1963).

The remaining 30 % of the plant feed was supplemented by underground mining that was established on the relatively narrow tin mineralised pegmatite bodies with stoping widths of ~3 m (up to 10 feet) and dips of 12° - 23°. Operations were conducted from shallow vertical and incline shafts, from levels spaced 30 m vertically. The mineralised material was blocked out by conventional development on reef, with raised connections at 100 m intervals on strike, depending on the tin grade (Anon., 1963).

Cassiterite was recovered from the mineralised material by gravity concentration after crushing. The mined material was mixed at the primary gyratory crusher which was set to give a 100 mm product and fed onto a 4,000 t open stockpile.

Secondary jaw crushers reduced the feed to 25 mm. This was followed by tertiary crushing using cone crushers and rod mills in closed-circuit with double deck screens giving a feed product of -2 mm. The cassiterite recovered from the separation screens was comprised two size fractions i.e., -2 mm to +1 mm and -1 mm, which were then treated in separate circuits. Historic recovery of the cassiterite was >75 % (Begg, 2008; Anon., 1963).

The mill concentrates, prior to smelting, were treated through magnetic separators to recover Ta/Nb pentoxides. The non-magnetic product was then smelted in an oil fired rotary furnace. The crude tin produced from the furnace was further refined in 10 t coke fired kettles and cast into ingots. The tin produced at Kamativi averaged 99.95 % pure which, at the time, was considered to be of an extremely high standard. In addition, a full range of solders and white bearing metals were manufactured and sold (Anon., 1963).

6.3 Historical Work on the Kamativi Tailings

The following summary relies on the information contained in the report by the United Nations Industrial Development Organization ("UNIDO") (UNIDO, 1983) which looked at the production of lithium chemicals in Zimbabwe. The UNIDO report (1982), makes reference to a report titled: "Pre-feasibility study for producing Lithium Chemical from Kamativi ores" (prepared for the Industrial Development Corporation of Zimbabwe Limited, May 1981). The UNIDO (1983) report provides a historical estimate of the size of the dumps (as of July 1980), the results of which are summarised in Table 6-3. The historical grade-tonnage estimate was based on a combination of sampling and limited drilling, however there are no detailed accounts of the drilling and sampling programmes that informed the historical estimate. The UNIDO (1983) report provides a summary of the results of chemical and mineralogical analyses (conducted in 1981) conducted on a composite sample of approximately 160 kg (360 lb) of dump material. The results are presented in Table 6-4 and Table 6-5. Sinclair (1996) reports a grade of 0.603 % Li_2O.

Table 6-3
Summary of the historical estimate for the lithium content of the tailings dump

	Tonnes	% Li_2O	Tonnes Li_2O	Tonnes LCE*
Grand Total	14,409,183	0.51	73,142	180,880 (1)

Note: ** LCE – Lithium Carbonate Equivalent*
[1] UNIDO (1983) indicates 160,904 t LCE but is an error as conversion to LCE (t) = Li_2O (t) x 2.473
Source: *UNIDO (1983)*

Based on the production figures presented in Section 6.1, these dumps were added to at a rate of approximately 0.9 Mt to 1.0 Mt per annum from 1984 until the mine closed in 1994.

Table 6-4
Summary of the chemical analysis of the main tailings dump

Element	Assay value (%)
Li_2O	0.70
Fe_2O_3	0.54
K_2O	2.52
Na_2O	5.12
Al_2O_3	15.17
SiO_2	72.7
MgO	0.51
LOI	1.21

Source: *UNIDO (1983)*

Table 6-5
Summary of the mineralogical analysis of the main tailings dump

Mineral	Assay value (%)
Muscovite	10.8
Spodumene	10.8
Sodium Feldspar	43.4
Potassium Feldspar	7.3
Quartz and other minerals	27.7

Source: *UNIDO (1983)*

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Source of the Tailings

The historic Kamativi tailings dump (Figure 7-1) is a man-made deposit that was generated as a site for the containment of tailings produced during the processing of tin mineralisation at the Kamativi Tin Mine, most recently owned by the ZMDC.

The Kamativi tailings were deposited over the period 1936 to 1994 and are derived from the mining and processing of the mineralised tin-bearing pegmatites.

7.2 Lithology and Mineralogy

The mineralisation of interest is contained in the dumps derived from the mining of the complex spodumene-bearing LCT pegmatites (ML No. 12 – Kamativi Mine). A summary of the main lithium minerals commonly associated with pegmatite hosted lithium deposits is presented in Table 7-1.

At Kamativi, spodumene is the main lithium mineral present with lesser amounts of cookeite, zinnwaldite, petalite and amblygonite (UNIDO, 1983). Historical estimates of the size and lithium content of the tailings dump (Table 6-5) indicate that there is lithium mineralisation of potential economic interest. Figure 4-3 shows the location of the tailings dump at the old Kamativi Tin Mine.

Table 7-1
Chemical composition and density of the main lithium minerals associated with pegmatites

Mineral	Chemical composition	Maximum* Lithium % (calculated)	Maximum* Li_2O % (calculated)	Density range (average)
Spodumene	$LiAl(Si_2O_6)$	3.7	8.0	3.15
Cookeite	$LiAl_4(Si_3Al)O_{10}(OH)_8$	1.33	2.86	2.67
Zinnwaldite	$K(Al,Fe,Li)_3(Si,Al)_4O_{10}(OH)F$	1.59	3.42	2.9-3.1 (3.0)
Petalite	$LiAl(Si_4O_{10})$	1.6-2.27	3.4-4.9	2.39-2.46 (2.42)
Amblygonite	$LiAl(PO_4)(F,OH)$	3.4-4.7	7.3-10.1	3.0
Lepidolite	$K_2(Li,Al)_{5-6}(Si_{6-7}Al_{2-1}O_{20})(OH,F)_4$	1.39-3.6	3-7.9	2.8-2.9 (2.84)
Eucrypite	$LiAl(SiO_4)$	2.1-5.5	4.5-11.8	2.67
Lithiophilite	$LiMnPO_4$	4.4	9.53	3.34

Note: * Note that the actual lithium content of the minerals may be lower and the numbers presented represent maximum theoretical lithium content.
Conversion factor from Li % to Li_2O % = Li % x 2.153

Source: *www.webmineral.com and BGS (2016)*

The pegmatites at Kamativi form part of a larger regional pegmatite belt within the Dete-Kamativi Inlier (Senzani, 1992) and are hosted in the supracrustal gneisses and schists of the Kamativi Schist Belt that form part of the Palaeoproterozoic Magondi Belt. The Magondi Belt comprises a dominantly metasedimentary succession with minor mafic, intermediate to felsic metavolcanics (Master, 1991) that was deposited between 2.1-2.0 Ga (Master, 1991) and has metamorphic ages of between 2.0-1.8 Ga (Master *et al.*, 2013). The Dete-Kamativi inlier is the western extension of the Magondi Belt exposed through younger Phanerozoic cover. The inlier comprises granodioritic orthogneisses, granites and highly deformed and metamorphosed supracrustal sequences that are divided in four northeast-southwest tectonostratigraphic belts. The four belts are known as the Kamativi, Tshontanda, Inyantue and Malaputese and overlie and are surrounded by the orthogneisses (Master, 1991; Senzani, 1992). The pegmatites have been dated at between 1026-925 Ma (Melcher *et al.*, 2013).

The pegmatites that were mined at the Kamativi Tin Mine are unzoned (Kinnaird *et al.*, 2016) bodies ranging from <1 m to >30 m wide and frequently bifurcate and pinch out along strike. Two types of pegmatite are recognised namely:

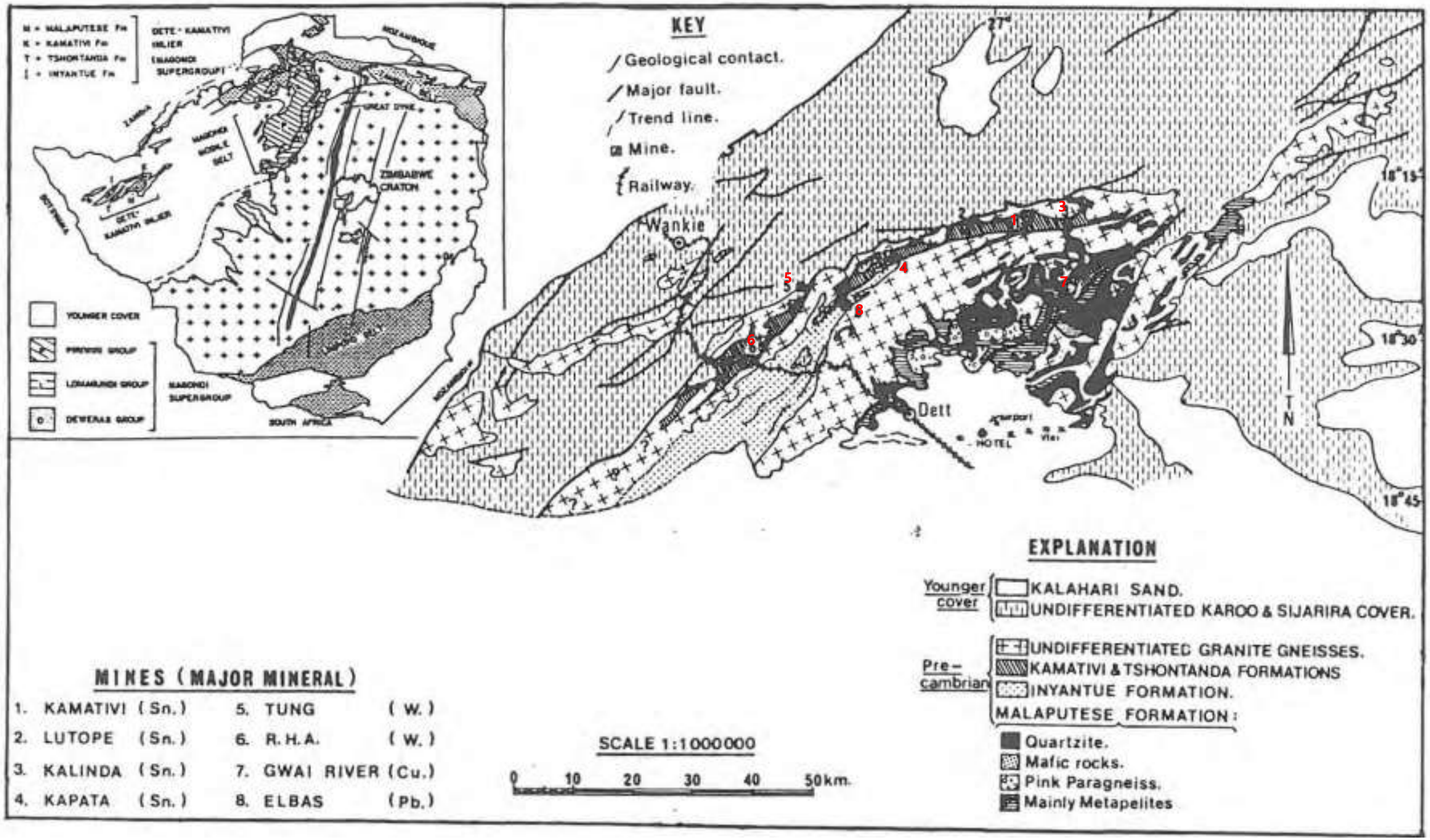
- stacks of thicker (up to 30 m+) inverted saucer shaped bodies and dips of between 20°-50° and have been traced for up to 2 km. These pegmatites occur within domal or anticlinal structures and are thickest in the centre of the domes/anticlines and thin outwards. The schist partings between the pegmatites may be up to 100 m thick. The pegmatite has been mined along a strike length of 800 m and to a depth of 270 m (Kinnaird *et al.*, 2016); and
- as discordant, foliation parallel dykes with dips of >70° and widths up to 8 m. These thinner pegmatite dykes occur less frequently and considered to be the feeders to the larger more common flat lying pegmatites (Senzani, 1992; Begg, 2008).

The pegmatite-schist contacts comprise a 10 cm thick wall-rock alteration zone comprising tourmaline, quartz and depleted in feldspar. The pegmatites are albitised and contain tin (in the form of cassiterite), lithium (in the form of spodumene and a variety of other lithium minerals) and tantalum/niobium (tantalite-columbite minerals) mineralisation (Senzani, 1992). The tin mineralisation occurs as cassiterite patches within the thinner pegmatite dykes and in the thicker flatter pegmatites it is more evenly disseminated with higher grade patches within individual pegmatites. The thicker (>4 m) pegmatites contain the lithium minerals, mainly spodumene which constitute up to 15% of the pegmatite composition. Average grades range from 0.65-0.75 % Li_2O and the spodumene contains 7 % Li_2O and <0.5 % Fe (Begg, 2008).

Wolframite is also reported to occur in small amounts in thin veins and stockworks of milky-grey quartz with minor tourmaline (Senzani, 1992).

Figure 7-1

Geological map of the Dete-Kamativi inlier.



Source: *Senzani (1992)*

8 DEPOSIT TYPES

The Kamativi tailings dump is a man-made deposit generated for the containment of tailings produced during the processing of the pegmatite-hosted tin ore at the Kamativi Tin Mine. The mined pegmatite material contains spodumene that was not recovered during the processing and is the lithium-bearing mineral of interest in this deposit.

The production history and records of tonnages and grades of the tin mineralisation mined as discussed in Section 6.

In order to understand the mineralogical composition of the tailings it is important that some background on the primary source of the lithium mineralisation is provided.

A pegmatite is defined as "an essentially igneous rock, commonly of granitic composition, that is distinguished from other igneous rocks by its extremely coarse but variable grain size or by an abundance of crystals with skeletal, graphic, or other strongly directional growth habits. Pegmatites occur as sharply bounded homogenous to zoned bodies within igneous or metamorphic host rocks." (London, 2008).

The main rock forming minerals in a granitic pegmatite include feldspar, mica (muscovite and biotite) and feldspar. Other minerals may occur in economic concentrations and include, but not limited, to various lithium minerals (Table 7-1), beryl, tourmaline, cassiterite, coltan, topaz, garnet and various rare-earth minerals.

Pegmatites are classified on the basis of a number of geological, textural, mineralogical and geochemical parameters and the accepted classification scheme. Pegmatites are broadly classified as either simple/common or complex based on the presence or absence of internal zonation. Simple/common pegmatites are unzoned, poorly fractionated and thus usually unmineralised. Complex pegmatites often contain potentially economic concentrations of mineral/elements (including Li, Ta, Nb, Sn, Be, REE) and their classification is based on a fourfold classification (Table 8-1) comprising:

- five classes namely abyssal, muscovite, muscovite-rare-element, rare-element and miarolitic classes, based predominantly on mineralogical and textural characteristics, the pressure and temperature conditions of pegmatite formation, and to a limited degree, the metamorphic grade of their host rocks; and
- the **classes** are further subdivided into **subclasses**, **types** and **subtypes** on the basis of geochemistry, mineral chemistry and mineral assemblages.

Further to the classification three broad pegmatite families are recognised based pegmatite classes to other petrological, paragenetic and geochemical data:

- **L**ithium-**C**aesium-**T**antalum (LCT);
- **N**iobium-**Y**ttrium-**F**luorine (NYF); and
- mixed LCT – NYF families.

Table 8-1

Pegmatite classification scheme of Černy and Ercit (2005) to illustrate the correlation between pegmatite classes and families

Class	Subclass	Type	Subtype	Family
Abyssal	HREE	-	-	NYF
	LREE	-	-	
	U	-	-	NYF
	BBe	-	-	LCT
Muscovite		-	-	
Muscovite-rare element	REE	-	-	NYF
	Li	-	-	LCT
Rare element	REE	allanite-monazite euxenite gadolinite	-	NYF
	Li	beryl	beryl-columbite beryl-columbite-phosphate	LCT
		complex	**spodumene** petalite lepidolite elbaite amblygonite	
		albite **albite-spodumene**	-	
Miarolitic	REE	topaz-beryl gadolinite-fergusonite	-	NYF
	Li	beryl-topaz spodumene petalite lepidolite	-	LCT

Note: *LCT = Lithium-Caesium-Tantalum; NYF = Niobium-Yttrium-Fluorine; see text for explanation*

It should be noted that pegmatites often occur as a combination/hybrid of the subtypes listed but with one or two of the minerals dominating over the other(s).

The Kamativi Pegmatites belong to the LCT family of pegmatites and can be classified as an unzoned Albite-Spodumene type and spodumene subtype of Complex type (Table 8-1), lithium subclass, rare-element class pegmatites (Černy and Ercit, 2005).

Rare-element pegmatites are often intruded into metamorphic rocks where the peak metamorphic conditions attained are upper greenschist to amphibolite facies (London, 2008) and have temporal and spatial associations with granitic plutons. Most pegmatites occur in swarms or pegmatite fields and occupies areas ranging from tens to hundreds of square kilometres; they may be associated with a discrete granite source around which they are systematically distributed, from the least fractionated granite to the most highly evolved pegmatites are the greatest distance from the granite source (London, 2008; Ercit, 2005); however, this is not always the case. With increasing fractionation, there is also often an increase in the complexity of the internal pegmatite zonation. The most highly evolved distal pegmatites are usually the most complexly zoned and associated with potentially economic concentrations of the elements and associated minerals identified above.

Pegmatites may vary from a few metres to hundreds of metres in length with variable widths ranging from <1 m to tens of metres wide and may have simple to complex internal structure. Cameron *et al*. (1949) identified up to nine different in internal units within a pegmatite based on differences in mineral assemblage, modes and textures which may or may not be present and/or continuous in a given pegmatite. These are summarised as follows (see also Figure 8-1):

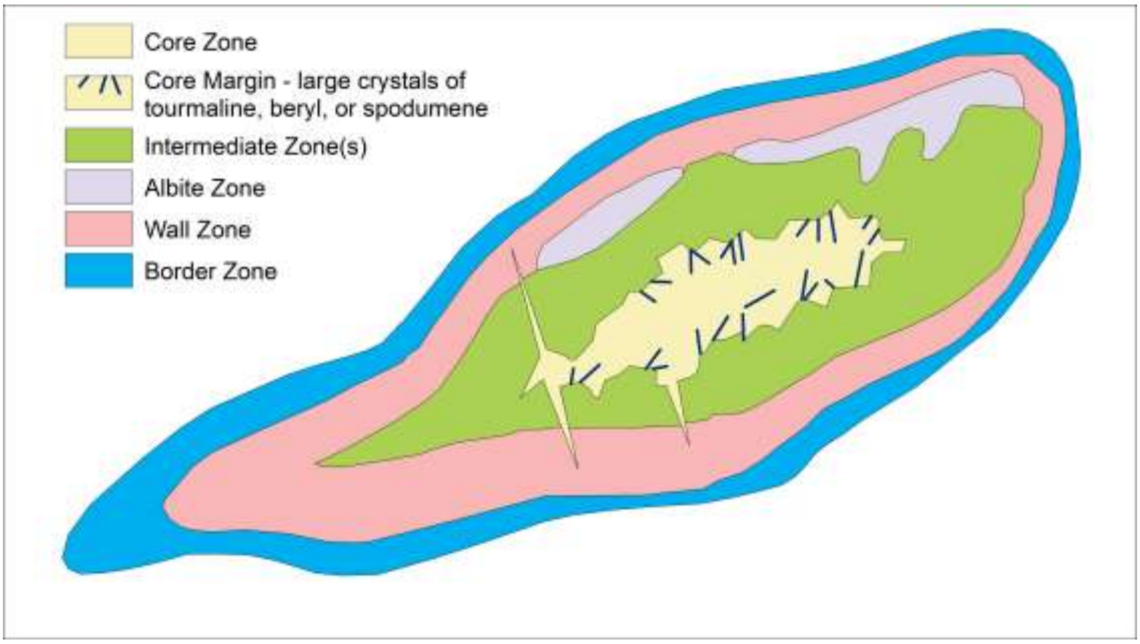
1. Zones of primary crystallisation forming more or less complete or incomplete concentric shells (asymmetric zonation also common) from the margin inwards:

    i. Border zone;
    ii. Wall zone;
    iii. Intermediate zones (outer, middle, inner and core margin); and
    iv. Core.

    With progressive crystallisation from the margin to the core, these zones usually display increasing grain size, decreasing number of rock-forming minerals, increasing number of accessory minerals and a change in texture from granitic or aplitic through graphic or heterogeneous in the border, wall and intermediate zones to blocky and coarse-grained monomineralic in the core (Cerný, 1991). It should be noted that this zoning is not always well developed and may be absent as is the case at pegmatites like Kamativi, Manono (in the DRC) and Arcadia (in Zimbabwe).
2. Replacement bodies form at the expense of pre-existing units with or without lithologic and/or structural control and are often difficult to identify as such. Their effects range from selective replacement of individual mineral species (e.g. micas after beryl or topaz), through to pervasive, yet diffuse, assemblages (e.g. albite and Li-mica after K-feldspar) replacing the primary minerals of an entire zone, to mappable, massive metasomatic units (e.g. massive lepidolite units and saccharoidal or platy albite units) replacing the bulk of the primary assemblage in pre-existing unit(s) (Cerný, 1991). These units can also contain potentially economic concentrations of cassiterite and columbo-tantalite.
3. Fracture fillings may be associated with primary zones or replacement units and are structurally controlled. These units are easily identified and generally insignificant. They are usually quartz-filled fractures emanating from the core and crosscutting the intermediate zones.

Figure 8-1

Schematic cross section of the internal structure of zoned pegmatites



Source: *After Cerný, 1991*

The economic mineralisation associated with pegmatites is usually associated with the intermediate, core margin and core zones and comprises mainly Li in petalite and lepidolite, Rb in K-feldspar and Cs in pollucite. Late stage replacement bodies comprising albite and lepidolite may also contain economic Nb-Ta, Li, Sn and Be mineralisation.

9 EXPLORATION

To date the exploration and test work on the Kamativi Project has been limited to the following:

- a review of the historical data for the Project (Project history presented in section 6);
- a topographical survey of the tailings dump;
- limited sampling on the top of the tailings dump; and
- drilling and sampling of the dump to inform a Mineral Resource estimate (see Section 13);
- XRD analysis of the samples from the drilling.

Mineral processing and metallurgical testing have also been done and a summary of this work is presented in Section 13.

9.1 Collar, Topographic Survey of the Tailings, Digital Terrane Model and Tailings Volume

A topographic survey was conducted by a registered qualified surveyor using a Geo survey portable differential GPS unit. The data was transferred to Harare-based resource mining consultants, Digital Mining Services, who constructed and generated a 3D Surpac volumetric model of the Kamativi dump. The dump volumes were subsequently calculated based on original surface topography digitised from the 1:50,000 scale topographic map (1827 A3, Kamativi, published in 1982) produced by the Zimbabwe Survey General.

The survey was completed by Mr Grabwell Fundira, who holds a National Diploma in Mine Surveying from the Zimbabwe School of Mines, Bulawayo. Mr Fundira used a Hi-Target V30 GNSS RTK GPS system comprising of a Base, Rover and External Radio and Controller. The Spatial Reference for the survey was UTM35K/ARC1950. All coordinates have been transformed to UTM35K/WGS84 for use in this report unless otherwise stated.

The survey took place in Matabeleland North Province, Zimbabwe. The type of equipment used is modern and accurate as it engages four types of satellite system, namely; GPS (American), GLONASS (Russian), Galileo (European) as well as BDS (Chinese).

A digital terrane model ("DTM") of the present-day surface was constructed using the data from the topographic survey and a pre-tailings DTM generated from the 1:50,000 topographic map (number 1827 A3, Kamativi, 1982). The pre-tailings DTM has however not been used for any of the volumes used in the Mineral Resource estimate.

Collar positions were surveyed in UTM35K/ARC1950 as drill holes were completed during the 2018 programme.

9.2 Sampling Results

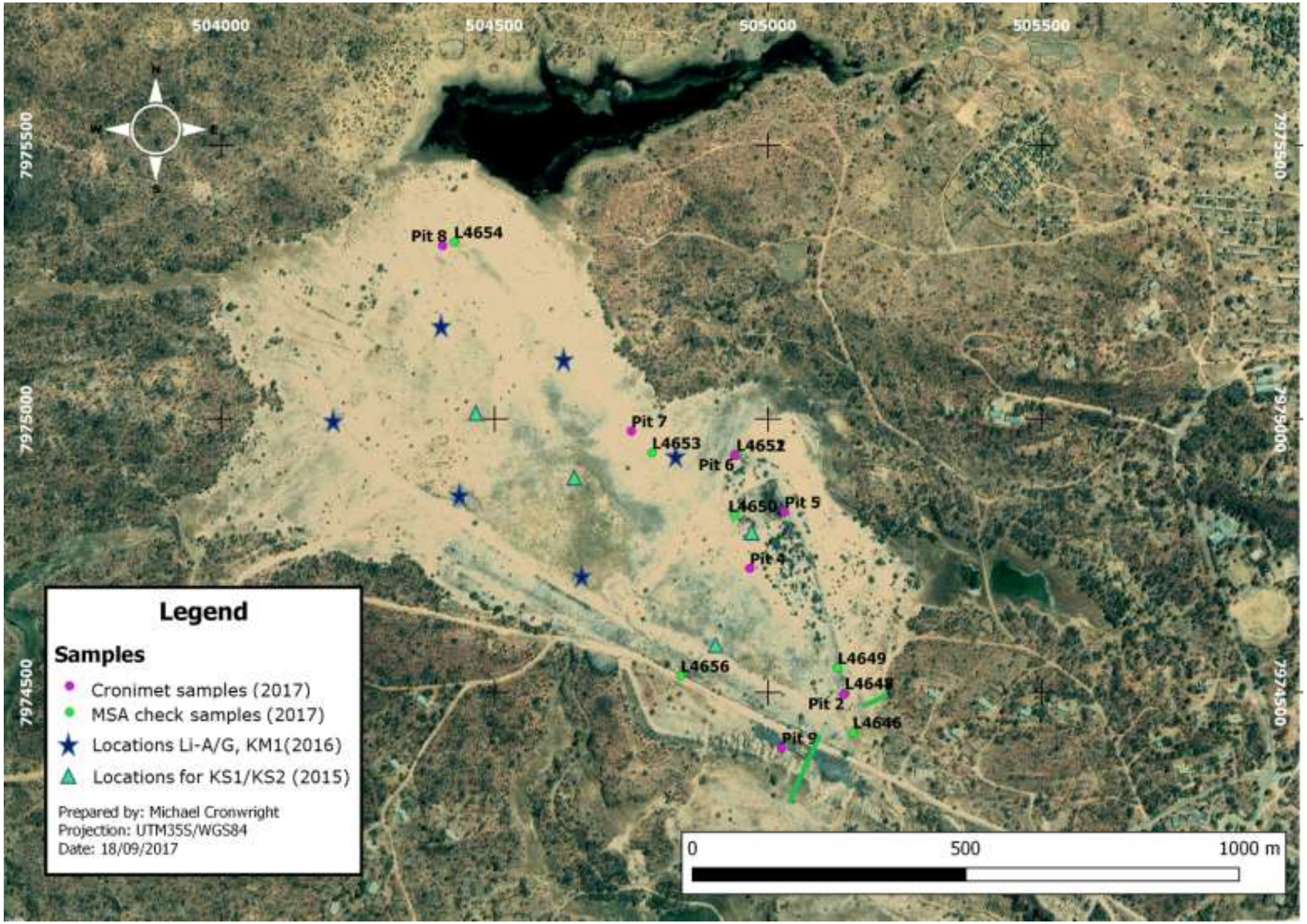
9.2.1 2015-2017 Sampling

Lintmar has conducted three sampling campaigns on the Kamativi tailings. The first sampling campaign was done in 2015, with follow-up sampling done in 2016 and 2017 which involved sampling of pits and trenches to a maximum depth of approximately 1.5 m. Table 9-1 provides a summary of the results (including the independent check sampling conducted by MSA – see also Section 11.3).

Composite samples KS1 and KS2, collected in the 2015 sampling campaign, comprise samples collected from six pits across the tailings dump; the positions were not recorded but the approximate positions are known and are shown in Figure 9-1. The material collected from the six locations was combined and two subsamples (KS1 and KS2) produced by cone and quartering. These two samples, KS1 and KS2, were used for initial mineralogical and metallurgical test work (Section 13).

Figure 9-1

Map with sample locations for MSA and historical sample locations. The locations for Li-A to G, KM1 and KS1 and KS2 are approximate.



Note: *Background map sourced from Google Earth (2018)*

The material for samples Li-A to Li-G was collected from seven different locations, comprising pits and trenches, across the tailings dump and then combined to form one sample; the positions where not recorded but the approximate positions are shown in Figure 9-1. The sampled material was then combined and split into seven subsamples, Li-A to Li-G, and submitted for assay to SGS South Africa (Pty) Ltd laboratory in Randfontein (Johannesburg) ("SGS Randfontein"). Sample material remaining from samples Li-A to Li-G after analysis by SGS Randfontein, was combined and split into samples, KM1 and the AH-series by SGS Randfontein for detailed test work currently on going at Axis House and the Mintek laboratories respectively (Section13).

As the sampling was not carried out systematically, the grades presented do not necessarily reflect the average grade of the tailings dump but do provide a range of Li_2O grades (Table 9-1 and Figure 9-2). These have been used to inform an Exploration Target which were reported in the previous Technical Report dated 4 April 2018. The results are comparable to Li_2O grades reported in the various references used in the compilation of this Technical Report. UNIDO (1983) reports grades of between 0.51 % and 0.705 % Li_2O and Sinclair (1996) reports a grade of 0.603 % Li_2O).

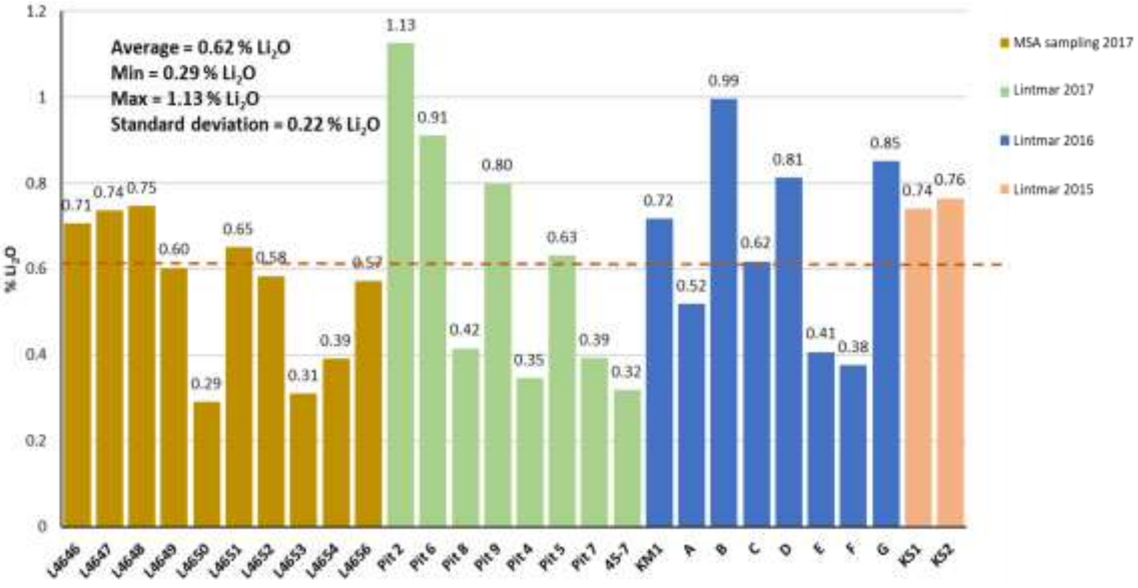
Table 9-1

Grab samples taken on the Kamativi tailings and results. Average grade is 0.62 % Li_2O (see Figure 9-1 and Figure 9-2)

Sample ID	Longitude	Latitude	% Li_2O	Year taken	Comments
L4646	-18.3198	27.04879	0.71	2017 (MSA check sampling)	
L4647	-18.3192	27.04865	0.74		
L4648	-18.3192	27.04865	0.75		
L4649	-18.3188	27.04852	0.60		
L4650	-18.3163	27.04674	0.29		
L4651	-18.3153	27.04677	0.65		
L4652	-18.3153	27.04677	0.58		
L4653	-18.3152	27.04531	0.31		
L4654	-18.3117	27.04188	0.39		
L4656	-18.3189	27.0458	0.57		
Pit 2	-18.3192	27.04863	1.13	2017	
Pit 6	-18.3153	27.04673	0.91		
Pit 8	-18.3118	27.04168	0.42		
Pit 9	-18.3201	27.04755	0.80		
Pit 4	-18.3171	27.047	0.35		
Pit 5	-18.3162	27.0476	0.63		
Pit 7	-18.3149	27.04495	0.39		
45-7			0.32		Combination of samples from pit 4, 5 and 7
KM1			0.72	2016	Used for Mineral Processing test work (see Section 1313)
Li-A			0.52		Used for Mineral Processing test work (see Section 1313)
Li-B			0.99		
Li-C			0.62		
Li-D			0.81		

Sample ID	Longitude	Latitude	% Li_2O	Year taken	Comments
Li-E			0.41		
Li-F			0.38		
Li-G			0.85		
Average KS1			0.74	2015	Used for Mineral Processing test work (see Section 13)
Average KS2			0.76		

Figure 9-2
Plot of all Li_2O results (see also Table 9-1)



These samples are not being used for Mineral Resource or Ore Reserve estimation, the procedures followed were considered adequate for the purposes of informing the Exploration Target.

9.3 Mineralogical Investigations

Fifteen composite samples from 7 of the auger drill holes were submitted to Geolabs Global in Centurion, South Africa for mineralogical analysis by XRD. The holes from which the samples were selected are spread across the surface of the dump (Figure 9-3). Sample material was taken from the samples submitted to SGS for assay and composited into samples ranging from 6 m to a maximum or 10.5 m.

The results of the XRD analyses (Geolabs Global, 2018) of the samples identifies spodumene as the dominant lithium mineral present (Table 9-2 and Figure 9-4); any other potential lithium mineral phases are below the XRD detection limit.

Five of the samples were screened to +425 µm and -425 µm fractions which were each separately analysed by XRD. These results are tabulated in Table 9-3. The results indicate that between 61 % and 92 % of the spodumene reports to the -425 µm fraction.

Figure 9-3

Plot of drill holes from which the samples were taken and analysed by XRD

Table 9-2

Results of the XRD analyses by Geolabs Global (assay normalised)

Hole ID	XRD ID	From (m)	To (m)	Length (m)	Sample ID	Li_2O (%)	Average Li_2O (%)	Mineral Abundance (%)									
								Quartz	Albite	Microcline	Muscovite	Clinochlore	Chlorite	Apatite	Spodumene	Cassiterite	Goethite
KT57	KT57-C1X	0	8	8	N4275	0.34	0.44	33.75	35.11	9.82	11.08	0.64	-	1.84	6.27	0.04	1.46
					N4276	0.46											
					N4278	0.54											
					N4279	0.57											
KT99*	KT99-C1X	0	9	9	N4456	1.06	1.08	37.67	41.54	4.94	8.08	-	0.2	0.78	6.33	-	0.45
					N4457	1.09											
					N4459	1.1											
	KT99-C2X	9	15	6	N4460	1.1	1.04	36.66	38.01	4.87	9.36	-	0.2	1.53	9.27	-	0.08
					N4461	0.99											
KT113	KT113-C1X	0	9	9	N4525	0.76	0.74	38.71	28.08	5.74	15.3	0.5	-	1.12	9.21	0.04	1.31
					N4526	0.79											
					N4527	0.67											
	KT113-C2X	9	18.31	9.31	N4529	0.64	0.65	35.73	32.8	6.61	13.37	0.57	-	1.11	8.03	0.05	1.72
					N4530	0.69											
					N4531	0.63											
					N4532	0.37											
KT27	KT27-C1X	0	9	9	N4164	0.46	0.45	37.42	31.81	7.31	14.41	0.57	-	1.48	5.58	0.05	1.37
					N4166	0.34											
					N4167	0.54											
	KT27-C2X	9	18	9	N4168	0.6	0.66	38.03	28.96	7.02	14.68	0.52	-	1.28	8.16	0.04	1.3
					N4169	0.63											
					N4171	0.73											

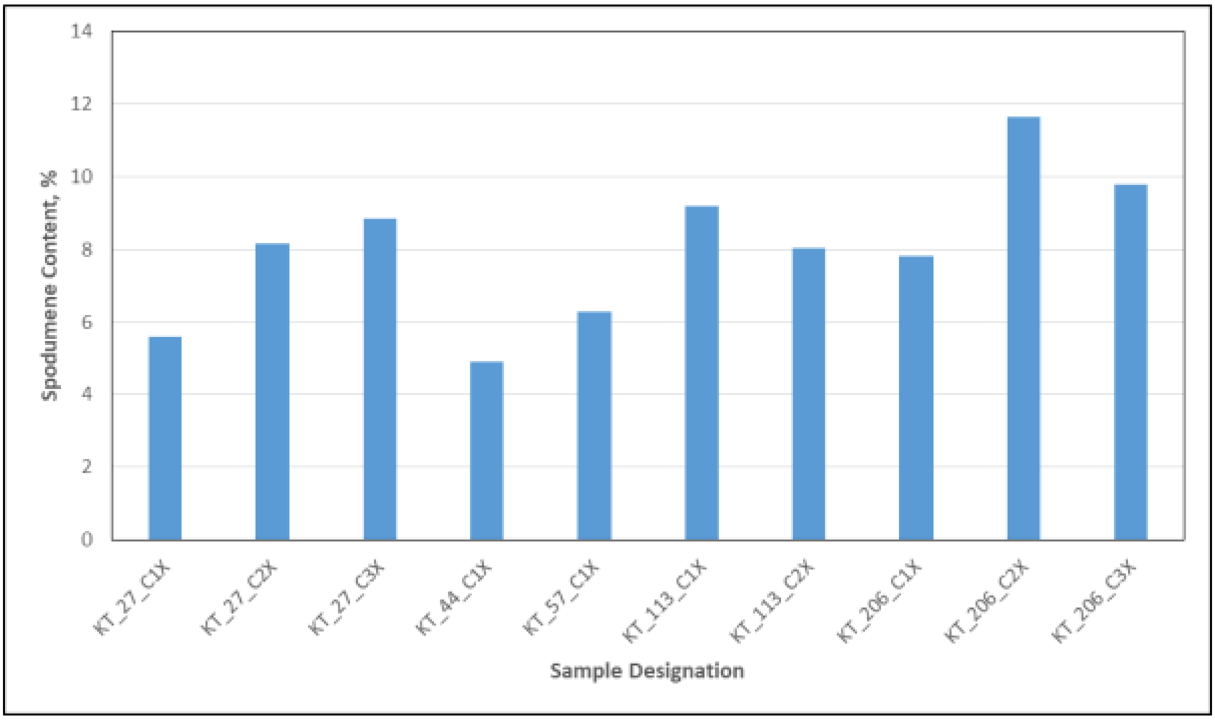
					N4172	0.73											
					N4173	0.69											
KT206	KT206-C1X	0	9	9	N4653	0.51	0.63	24.39	37.41	6.4	19.89	0.7	-	1.9	7.82	0.05	1.45
					N4655	0.56											
					N4656	0.81											
	KT206-C2X	9	18	9	N4657	0.89	0.94	29.07	34.22	6.3	15.27	0.54	-	1.54	11.64	0.05	1.37
					N4658	0.94											
					N4659	0.97											
	KT206-C3X	18	24.48	6.48	N4661	0.83	0.79	28.59	38.58	5.81	13.94	0.48	-	1.41	9.79	0.04	1.36
					N4663	0.71											
					N4664	1.03											
KT210*	KT210-C1X	0	9	9	N4697	0.38	0.46	40.53	39.19	7.42	8.21	-	0.3	0.52	3	-	0.84
					N4698	0.43											
					N4699	0.57											
	KT210-C2X	9	18	9	N4701	0.51	0.54	38.69	40.01	7.87	7.61	-	0.27	0.4	3.83	-	1.32
					N4702	0.56											
					N4703	0.54											
	KT210-C3X	18	28.5	10.5	N4704	0.55	0.51	37.66	42.29	7.05	8.37	-	0.2	0.56	3.72	-	0.05
					N4706	0.51											
					N4707	0.47											
					N4708	0.51											
KT44	KT44-C1X	0	9.8	9.8	N4230	0.44	0.39	17.79	47.7	4.4	19.29	2.05	-	2.08	4.9	0.05	1.75
					N4232	0.37											
					N4233	0.37											
					N4234	0.39											

Table 9-3

Results of the XRD analysis of the size fractions by Geolabs Global

Sample ID	Spodumene Recovery (%)	Mass (%)	Quartz (%)	Albite (%)	Microcline (%)	Muscovite (%)	Chlorite (%)	Apatite (%)	Spodumene (%)	Goethite (%)
KT99_C1X +425	28.07	27.75	46.7	32.8	7.4	4.4	0.2	1	6.4	1.1
KT99_C1X -425	71.93	72.25	34.2	44.9	4	9.5	0.2	0.7	6.3	0.2
KT99_C1X Calculated		**100**	**37.67**	**41.54**	**4.94**	**8.08**	**0.2**	**0.78**	**6.33**	**0.45**
KT99_C2X +425	31.96	26.68	45.6	28.7	6.7	6.5	0.2	0.8	11.1	0.3
KT99_C2X -425	68.04	73.32	33.4	41.4	4.2	10.4	0.2	1.8	8.6	0
KT99_C2X Calculated		**100**	**36.66**	**38.01**	**4.87**	**9.36**	**0.2**	**1.53**	**9.27**	**0.08**
KT210_C1X +425	38.13	38.13	46.9	33	8.9	5.3	0.3	0.4	3	2.2
KT210_C1X -425	61.87	61.87	36.6	43	6.5	10	0.3	0.6	3	0
KT210_C1X Calculated		**100**	**40.53**	**39.19**	**7.42**	**8.21**	**0.3**	**0.52**	**3**	**0.84**
KT210_C2X +425	29.02	28.49	45.7	29.5	9.3	9.4	0.2	1.4	3.9	0.6
KT210_C2X -425	70.98	71.51	35.9	44.2	7.3	6.9	0.3	0	3.8	1.6
KT210_C2X Calculated		**100**	**38.69**	**40.01**	**7.87**	**7.61**	**0.27**	**0.4**	**3.83**	**1.32**
KT210_C3X +425	7.92	12.26	43.8	30.8	6.7	14.6	0.2	1	2.4	0.4
KT210_C3X -425	92.08	87.74	36.8	43.9	7.1	7.5	0.2	0.5	3.9	0
KT210_C3X Calculated		**100**	**37.66**	**42.29**	**7.05**	**8.37**	**0.2**	**0.56**	**3.72**	**0.05**

Figure 9-4
Abundance of spodumene in selected auger samples by assay validated Rietveld Refinement XRD (see also Table 9-2)



9.4 Density Pits

A total of 60 pits were excavated across the dumps for density measurements of the material close to the surface below the loose surface material, but where the material was consolidated. Pits were excavated at 1m depth increments (Figure 9-5) with depths ranging from 1 m up to 6 m.

Once a pit had been excavated, samples were collected from the sidewall of the pit. The samples were collected as follows:

1) A square tube of 10 cm x 10 cm x 10 cm dimensions was knocked into the top of sidewall of the pit and to be flush with the surface of the sidewall. This was done where the material had not been disturbed.

2) Using the trowel or small spade the material within the tube was carefully removed and caught in the catch-pan or square plastic container.

3) The material was then transferred to a sample bag and weighed

4) The sample ID was recorded as KTDPXX-YY where XX was the pit ID and YY the sample increment from where the sample was taken in the pit. The sample depth was also recorded. For example - the sample increment starts at .01 for the first sample. Then .02 would be the next sample. All this information was captured onto a log sheet and then transferred to a Microsoft Excel spreadsheet.

5) The process is then repeated at 1 m increments to the base of the pit

6) All samples were then to be air dried and weighed again.

The in-situ density and air-dried density was then calculated for each sample as follows: **In situ density in kg/dm^3**= sample mass (as recovered) (kg)/volume of steel pipe (dm^3). Note that the square tube of internal dimension 10x10x10 cm = 1000 cm^3 = 1 dm^3 = 1 litre.

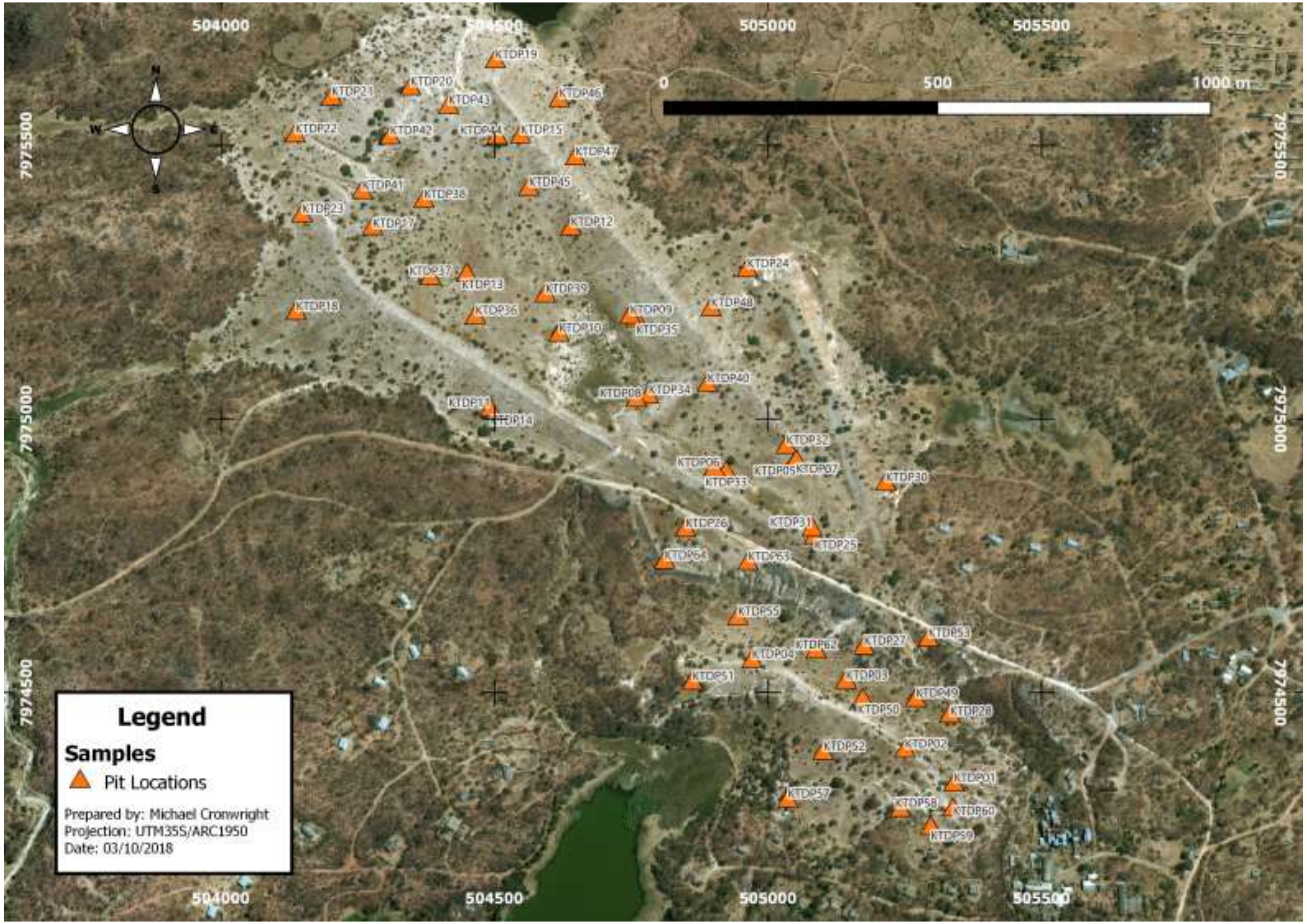
Air dried density in kg/dm^3 = sample mass (air dried) (kg)/volume of steel pipe (dm^3) The moisture content of the sample was also determined by the following calculation: % moisture by mass= 100*(in situ mass-air dried mass)/in situ mass. Table 9-4 summaries the density and moisture content values.

Table 9-4
Summary of in situ and dry density values from density pits (n=213 from 60 pits)

	In Situ Density (kg/dm^3)	Dry Density (kg/dm^3)	Moisture Content (%)
Average Value	1.77	1.67	5.94
Minimum Value	1.15	1.09	0.21
Maximum Value	2.18	2.08	22.89

Figure 9-5

Map with sample locations of density pits



9.5 Exploration Target

As part of the earlier work done, an Exploration Target was reported in the previous Technical Report dated 3 April 2018.

10 DRILLING

Exploration drilling was focussed on the tailings dump derived from the historical tin mining conducted at the now disused Kamativi Tin Mine.

A total of 115 drill holes were drilled between 2 March 2018 and 29 May 2018 across the tailings dump to define the extent and depth of the dump. One hole was abandoned due too excessive collapsing and the samples were not submitted for assay. The abandoned hole was re-drilled. Drilling was carried out using an auger drill and for a portion of the programme a diamond coring rig. The split of the holes drilled between the two methods are summarised in Table 10-1.

Table 10-1

Summary of drilling conducted by MSA on the Property in the current phase of exploration

Phase of exploration	Type	Number of drill holes	Drilled metres by diameter (m)		
			PQ (m)	HQ (m)	NQ (m)
2018 Exploration Programme	DD	60	450.51	169.44	82.41
	Auger	55	1,172.62		
Total Metres Drilled	**1874.98**				

The diamond drilling was done using a single HZ–150Y rig using a combination of NQ, HQ and PQ bit sizes with internal diameters of 60 mm, 76 mm and 105 mm. All drilling was done without a core barrel and dry (no drilling fluids or water was used) and drill runs of 1.5m were drilled in order to minimised loss of material. The material was of sufficient fine-grained nature and moisture content for the holes to remain open and re-enter for each drill run, as well remain in the core barrel and recovered for sampling purposes. A depth check was conducted on re-entry to the hole and where either loss of material had occurred, or the hole collapsed; if this was the case the material was recovered and discarded. A number of holes were either abandoned and stopped short of the base of the dump due the holes continuously collapsing or the inability of the drill rig to progress. The samples collected from the 1.5 m runs were large (up to 45 kg and averaging 18 kg) and thus had to be split into multiple bags.

The auger drilling was done using power auger drilling rig (Figure 10-1). These portable rigs utilise a rotating spiral auger encased in a stainless-steel core barrel to advance into the tailings dump. This method ensures that contamination of the sample is minimised. Because the drilling is performed dry, under relatively stable conditions, no back-mixing of the samples or chemical alteration of the elements in the samples can occur. The problems traditionally encountered with drilling muds are thus eliminated. The auger is advanced to refusal ("hard objects") or until soil is encountered, indicating that the base or natural ground level below the tailings has been reached.

The rig utilises a 50 mm nominal bore drill rod producing approximately 3.5 kg of sample per metre drilled. Samples are drawn in 1.5 m increments and approximately 5 kg of sample recovered per increment.

As the hole depth increases, extension rods are connected to the core barrel.

The rods and core barrel are withdrawn from the drill hole by a hydraulic driven extractor box.

The extension rods and spiral augers (core barrel) are in three lengths, namely 1.5 m, 3.0 m and 4.5 m. This allows samples to be extracted in 1.5 m intervals. A typical cycle would be:

- Drill with the 1.5 m spiral auger
- Drill with the 3.0 m spiral auger
- Drill with the 4.5 m spiral auger
- Drill with the 4.5 m spiral auger + 1.5 m extension
- Drill with the 4.5 m spiral auger + 3.0 m extension
- Drill with the 4.5 m spiral auger + 4.5 m extension
- Drill with the 4.5 m spiral auger + 4.5 m + 1.5 m extension

The last step is continued until soil is encountered which indicates the bottom of the hole. The hole is thus only advanced by 1.5 intervals per sample collection.

On withdrawing of the core barrel, the extension rods are removed, and the core barrel is placed on two tripods. These tripods are placed on top of a 1.2 m x 5.0 m sheet of plastic to ensure total sample collection.

The spiral is first rotated by hand, in the opposite direction to ‘loosen’ the sample. Some of the sample is collected at the end of the core barrel due to the redirection of the rotation. The spiral is removed from the core barrel and cleaned. The sample is gathered from the plastic sheet and the material is bagged in a strong marked plastic bag, before cleaning the plastic sheet. The soil collected from the bottom of the drill hole is kept and bagged separately.

Figure 10-1
Power auger drilling

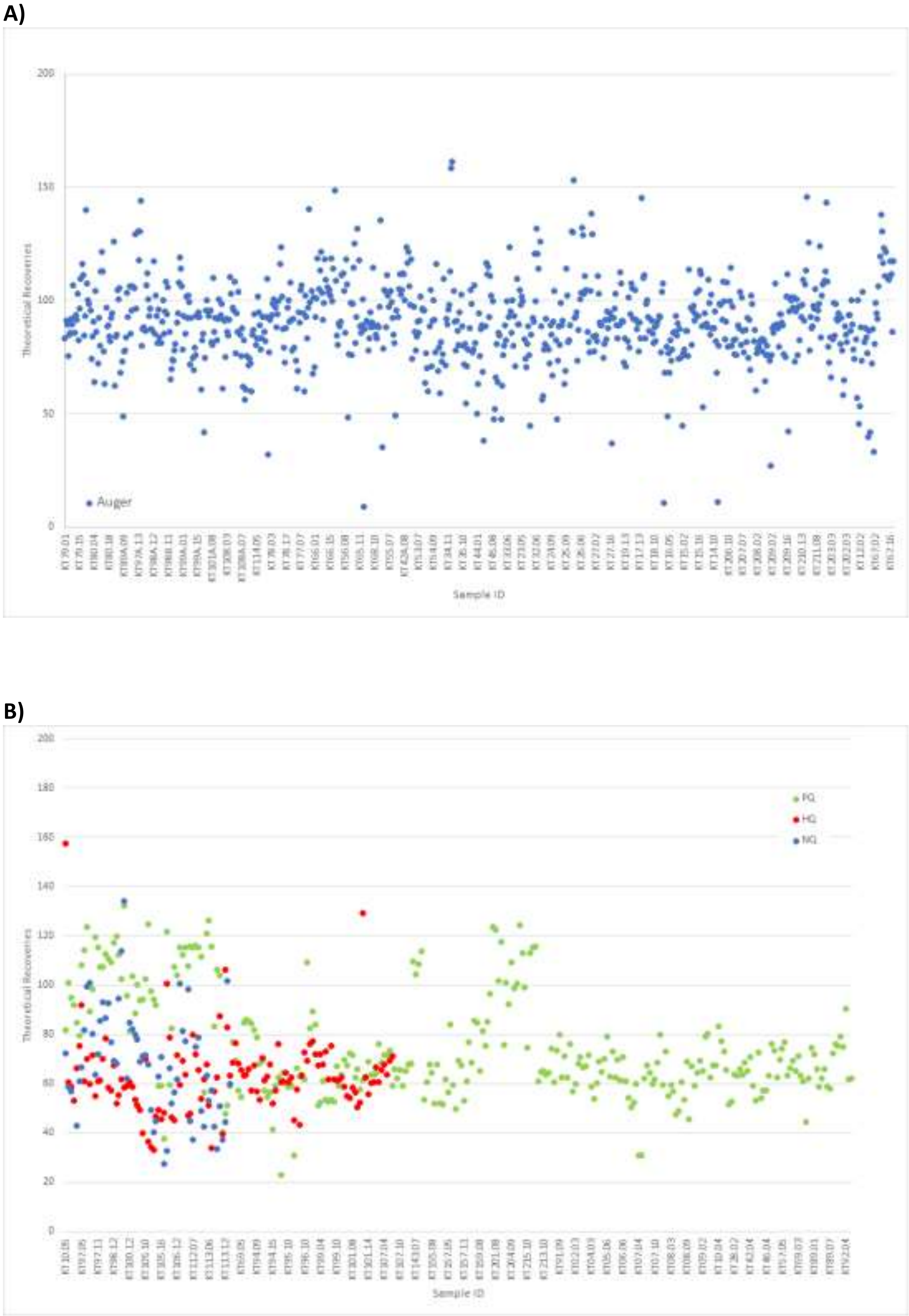
10.1 Sample Recoveries

Sample recoveries for auger drill samples show a high degree of consistency. However, there are outliers which can be attributed to hole collapse and core loss resulting in variable gains and losses for some samples (Figure 10-2A).

The recovery of the samples from the diamond drill holes is more variable (Figure 10-2B) as these holes were more prone to collapsing and losing some of the sample when tripping the rods at the end of a 1.5 m drill run. At the end of each drill run the drill rods had to be completely removed from the hole when retrieving the sample.

Figure 10-2

Distribution of sample recoveries for A) Auger and B) the DD drill samples



For the diamond drill samples, sample recoveries were calculated as follows:

1. A theoretical sample mass was calculated as an in situ sample masses using the external diameter of the drill rods (i.e. NQ=71 mm; HQ=92 mm and PQ=128 mm), an in situ density calculated from the density pits of 1.77 kg/dm^3 (see Section 9.3) and the length (1.5 m) of the sample recovered.
2. The recovery was calculated as follows: (in situ sample mass recovered)/(theoretical in situ sample mass)x100

The recoveries for the auger drill samples were calculated in the same way, however instead of using a wet mass, a dry mass was used, which was reported by the laboratory. Summary statistics of the recoveries are provided in Table 10-2.

Table 10-2
Summary statistics of theoretical sample recoveries for Auger and DD holes

	Auger	**DD**		
Outer Diameter	**50mm (n=788)**	**NQ – 71mm (n=63)**	**HQ – 92mm (n=124)**	**PQ – 128mm (n=297)**
Average (%)	90.6	71.6	63.2	75.0
Minimum (%)	8.8	27.5	33.2	23.1
Maximum (%)	161.2	278.3	157.4	132.3
Overall Average Recoveries (%)	90.6	71.6		

10.2 Conclusion

Based on the drilling and sampling conducted in the current phase of exploration the QP is of the opinion that the sampling is representative of the mineralization present.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Previous sampling campaigns

A total of 28 samples have been collected across the tailings dump comprising 18 samples collected in 2015, 2016 and 2017 by Lintmar and another 10 samples collected for verification purposes by MSA in 2017 (see Section 12.1) as listed in Table 9-1. All samples were sent to SGS Randfontein for sample preparation and assay. SGS Randfontein is accredited by SANAS (South African National Standard) and conforms to the requirements of ISO/IEC 17025 for the analytical methods used as per Table 11-1.

A number of sample splits, from samples Li-A to Li-G and KM1, as described below were also sent to various laboratories (Axis House in Cape Town and Mintek in Randburg, Johannesburg) for metallurgical test work, the details of this work is described in Section 13.

Table 11-1
Summary of assay methods used by SGS Laboratories for the verification sampling (for 2018 drilling campaign see also Figure 11-3)

Method code	Elements	Lower detection limit	Upper detection limit	Description
PREP method	Sample dried and 500 g split by riffle splitter. The 500 g split of 2 mm material pulverized to 85 % passing 75 micron in a Carbon Steel ring and puck pulverizer.			
IC90A	Li Fe Other elements*: Al,Ba,Ca,Cr,Cu,K,Mg,Mn,P,S,Si,Sr,Ti,V,Zn	10 ppm 0.01 %	100,000 ppm 30 %	Sodium peroxide fusion, ICP-OES finish
IC90M	Sn Other elements*: As,Be,Bi,Cd,Ce,Co,Cs,Dy,Er,Eu,Ga,Gd,Ge, Ho,In,La,Lu,Mo,Nb, Nd,Ni,Pb,Pr,Rb,Sb,Sc,Sm,Sn,Ta,Tb,Th,Tl, Tm,U,W,Y	1 ppm	10,000 ppm	Sodium peroxide fusion, ICP-MS finish

Source: ** - detection limit varies depending on element*
*** Sodium peroxide fusion, ICP-OES, ICP-MS Finish*

The samples were collected from the sides of the tailings dump and trenches and pits excavated into the top of the tailings dump. The Pit-series and MSA check samples were collected as continuous channel samples taken from the side of each excavation using a shovel (Figure 11-1). Each sample was put into a pre-labelled sample bag and sealed with a cable tie.

Samples KS1 and KS2 collected in 2015 from the tailings and submitted for particle size distribution ("PSD") and grade by size fraction as SGS Randfontein. The Li_2O grades reported in Table 9-1 for these samples are calculated weighted average grades.

In 2016 samples Li-A to Li-G were collected and a subsample of each assayed at SGS Randfontein. The remainder of the material for samples Li-A to Li-G was then combined and split into the AH-series of samples that were submitted to Axis House for metallurgical test work which is currently ongoing and KM1 which was sent to Mintek for metallurgical test work. Details of the test work conducted are discussed in detail in section 13 of this Technical Report.

No independent quality assurance and quality control ("QA/QC") protocol was implemented by Lintmar for any of the sampling conducted to date. SGS Randfontein routinely implemented QA/QC using replicate analysis and internal reference materials.

Figure 11-1
Sampling of pit wall using a shovel

The sample number and location were noted by the geologists and technicians on hardcopy and then recorded in a Microsoft Excel spreadsheet as seen in Table 9-1 .

All samples were transported from site to South Africa by Lintmar personnel.

11.2 2018 Borehole sampling, preparation and analysis

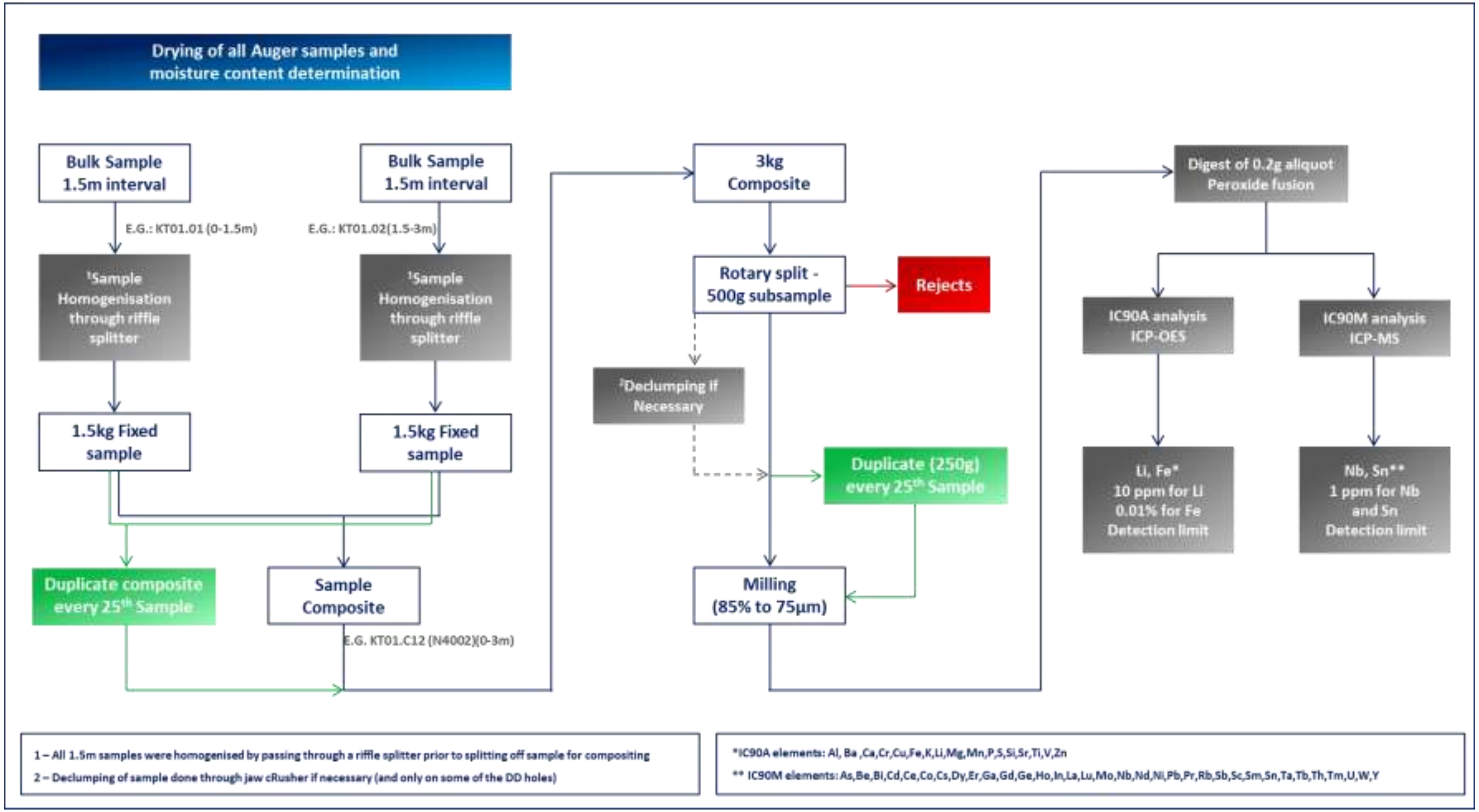
After each 1.5 m drill run, the samples were bagged into heavy duty sample bags and laid-out by the drillers in the sequence that they were drilled – this process was supervised by the on-site geologist. The sample bags containing the 1.5 m samples were labelled clearly with a paint marker by the on-site geologist – indicating both the borehole ID and the drilled interval on each sample bag (Figure 11-2). A sample ticket number was also inserted into each sample bag. All sample masses were recorded on site to three decimal places. The scale used to record masses was regularly calibrated to ensure accurate measurements.

Figure 11-2
Sample collection with borehole ID and interval during power auger drilling

A total of 1,272 samples were collected at 1.5 m lengths in this phase of the exploration drilling campaign and bagged and sealed on site. Samples were kept on site until inspected and signed off by Zimbabwe Ministry of Mines and then transported to a storage depot in Bulawayo where they were kept until the end of the drilling programme. At the end of the programme the samples were

Figure 11-3

Sample preparation flowchart showing the sample preparation, compositing and the preparation of the sample duplicates and analysis.



1 – All 1.5m samples were homogenised by passing through a riffle splitter prior to splitting off sample for compositing

2 – Declumping of sample done through jaw cRusher if necessary (and only on some of the DD holes)

*IC90A elements: Al, Ba ,Ca,Cr,Cu,Fe,K,Li,Mg,Mn,P,S,Si,Sr,Ti,V,Zn

** IC90M elements: As,Be,Bi,Cd,Ce,Co,Cs,Dy,Er,Ga,Gd,Ge,Ho,In,La,Lu,Mo,Nb,Nd,Ni,Pb,Pr,Rb,Sb,Sc,Sm,Sn,Ta,Tb,Th,Tm,U,W,Y couriered to SGS Randfontein Johannesburg, South Africa for sample preparation and assay. Figure 11-3 summarises the sample preparation procedure from compositing of the samples through to assay. The analytical methods used are as per Table 11-1. A QP visit to the lab was done on 22 June 2018.

All samples were composited to 3 m intervals by combining equal masses of material from two successive 1.5 m samples down the drill holes. Where the last sample in a drill hole did not represent a 1.5 m interval or could not be composited with another to make a 3 m composite the sample was left uncomposited.

11.2.1 Quality Assurance and Quality Control Procedures

In addition to the laboratory QA/QC programme routinely implemented by SGS using pulp duplicate analysis and internal blank and standards, MSA implemented an external QA/QC protocol comprising the insertion of certified reference materials ("CRMs"), blanks and composite duplicates, and sample split duplicates on a systematic basis amongst the samples shipped to SGS (discussed below). These were inserted at a frequency of 1 blank, 1 CRM and 1 duplicate for every 25 samples (giving an average of approximately 12 %) (Table 11-2).

Table 11-2
Summary of the QC samples inserted into sample stream

Sample Type	Assays reported	Percentage (%)
Composite Duplicates	29	4.2
Sample Duplicate	27	3.9
AMIS0338	28	4.0
AMIS0341	28	4.0
AMIS0484 (Blank silica powder)	36	5.2
Total QA/QC Samples	**148**	**21.3**
Lab Check samples	**41**	**6**
Samples	693	**27.3**
Total	**882**	

11.2.2 Certified Reference Materials

Two different CRMs were used for the internal QA/QC programme, namely AMIS0338, a low grade lithium CRM, and AMIS0341, a high-grade lithium CRM relative to the deposit grades. Both were sourced from African Mineral Standards ("AMIS") in Johannesburg, South Africa. Table 11-3 provides a summary of the expected values for the CRMs.

Table 11-3
Summary of the certified lithium values for the CRMs used

CRM	Element	Method	Expected value	Two standard deviations
AMIS0338[1]	Li	Fusion	1,742 ppm	±439 ppm
AMIS0341[1]	Li	Fusion	5,041 ppm	±222 ppm

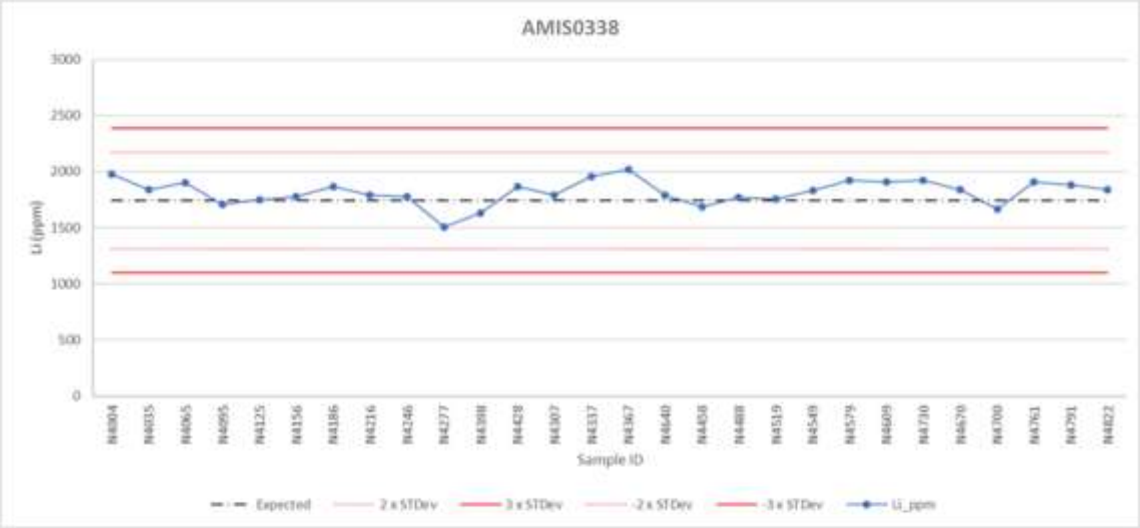
Note: **1-** *The complete certificates can be downloaded from http://www.amis.co.za/certificates-products-in-stock*

CRMs AMIS0338 and AMIS0341 each make up approximately 4 % of the total samples submitted i.e. a CRM at every 25th sample was inserted into the sample stream. Plots of performance of the AMIS0338 and AMIS0341 CRMs are shown in Figure 11-4 (A and B). The assays for AMIS0338 report within the 2x standard deviation ("SD") warning limits and are considered acceptable. Several assays for AMIS0341 fell outside the 2xSD warning limit (and three fell outside the 3xSD limit) but are not considered material to the reported results and overall the performance of AMIS0341 is considered acceptable.

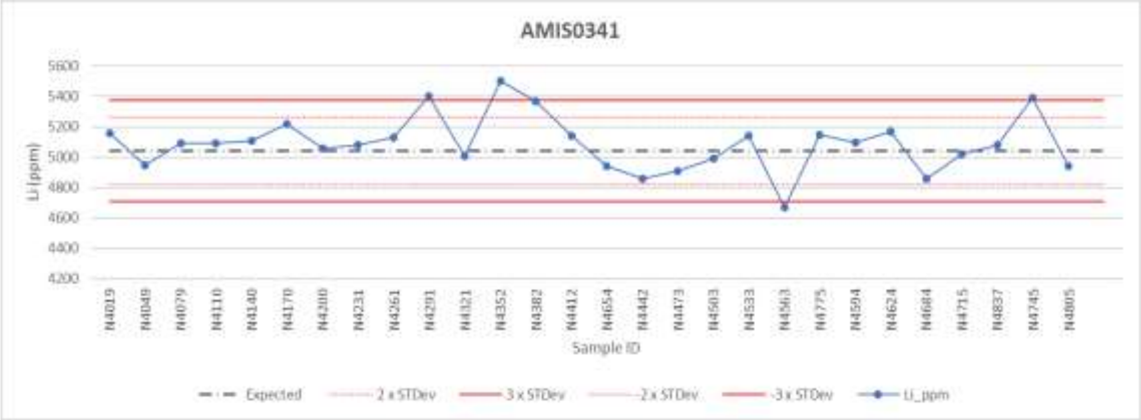
Figure 11-4

Performance of A) AMIS0338 (low grade, n=28) and B) AMIS0341 (high-grade, n=28) through the current exploration programme for all samples assayed

A



B



11.2.3 Blanks (AMIS484 – blank silica powder)

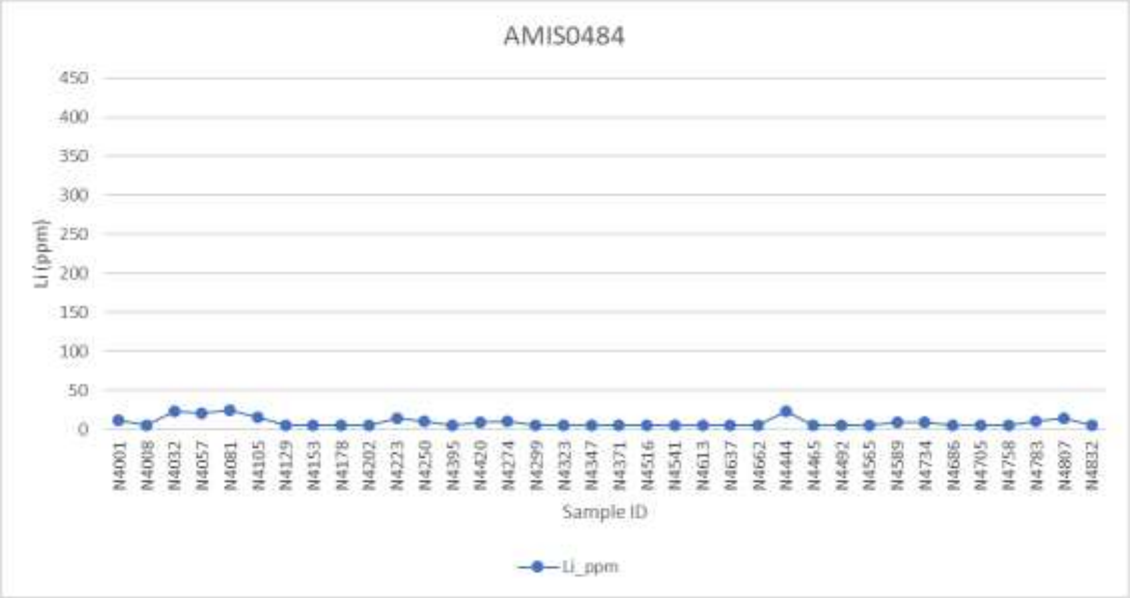
Quartz blanks were inserted into the sample stream as part of the external QA/QC protocol. AMIS0484, a blank silica powder, was used and inserted at a frequency of one in every 20 samples comprising 5 % of the sample stream.

Figure 11-5 plots the performance of the blank material used throughout the current exploration programme. AMIS0484 reports Li values between 5 and 25 ppm showing minor cross contamination. However, the level of these Li values is considered immaterial and the results are acceptable.

Figure 11-5

Performance of AMIS0484 (blank silica powder, n=36)

through the current exploration programme for all samples assayed



11.2.4 Sample Duplicates

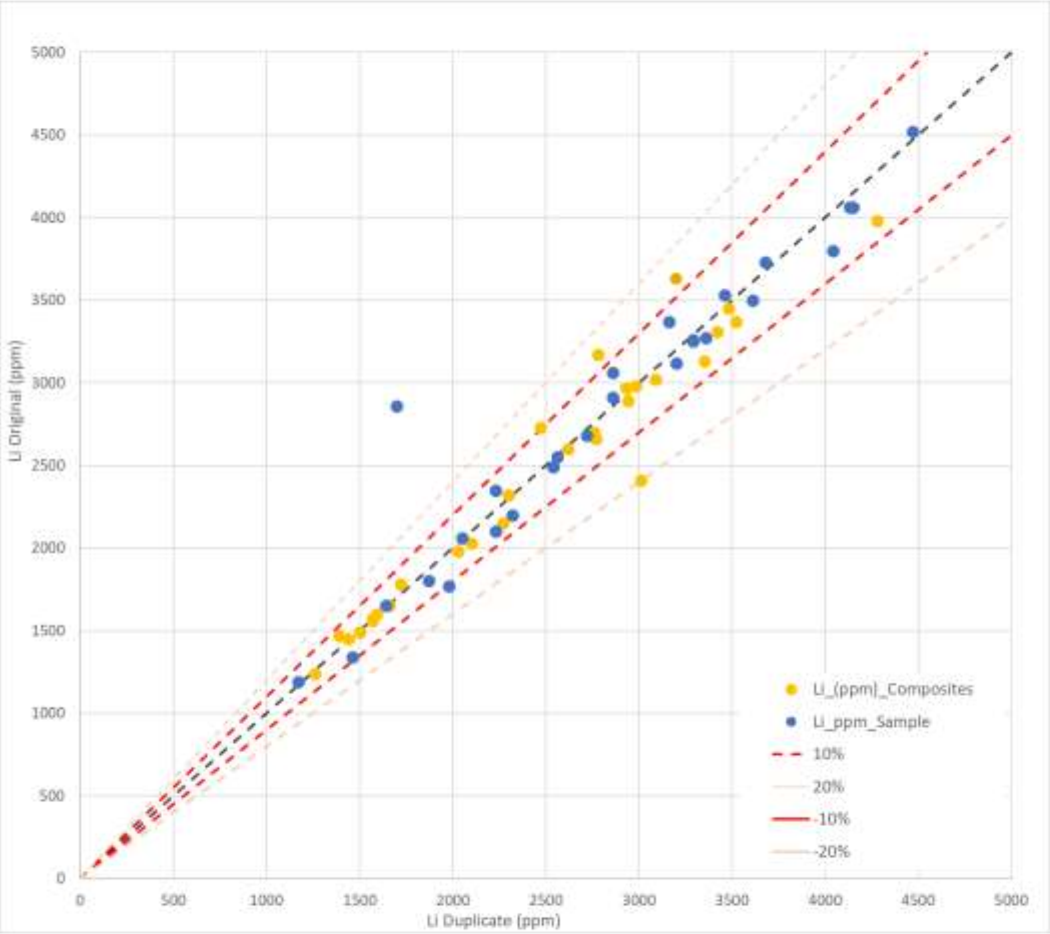
The sample preparation flowchart as shown in Figure 11-3 illustrates the methodology that was followed when creating duplicates. Two different duplicates were prepared and analysed.

Approximately 4 % of the samples submitted to SGS were made up of composite duplicates where material from two consecutive 1.5 m intervals were homogenised. The two splits of each of the 1.5 m intervals were taken to create two separate composites, namely a primary composite and a composite duplicate. The composite duplicates were submitted at a frequency of one in 25 into the sample stream and confirmed the consistency of the compositing procedure and analytical methodologies. The duplicates correlate well with the original composite samples (Figure 11-6). Two samples fall well outside the 10% variance line and one sample falls between the 10 and 20% variance line. Overall, the results for the composite duplicates are considered acceptable.

Sample duplicates of 3 m composite pulps were also submitted to the laboratory at a frequency of one in every 25 samples comprising 4% of the sample stream. The duplicates were split after crushing of samples and prior to pulverising. The duplicates correlate well with the original samples, with the exception of one sample where the original reports higher than the duplicate. Despite one outlier, the overall comparability is considered acceptable. Figure 11-6 shows both composite duplicates and duplicates of the composite pulps plotted against the original samples.

Figure 11-6

Plot of duplicates vs original samples for current exploration programme (n=27) and original composites vs duplicate composites (n=29)



11.2.5 Check Laboratory Samples

A total of 51 samples were submitted to ALS Chemex in Vancouver for check analyses by ME-MS89L (Table 11-4). The check samples submitted consisted of 41 samples representing 6 % of the total number of samples, together with four AMIS484 and six AMIS338 CRMs. Figure 11-7shows the original sample results plotted against the check sample results.

Table 11-4

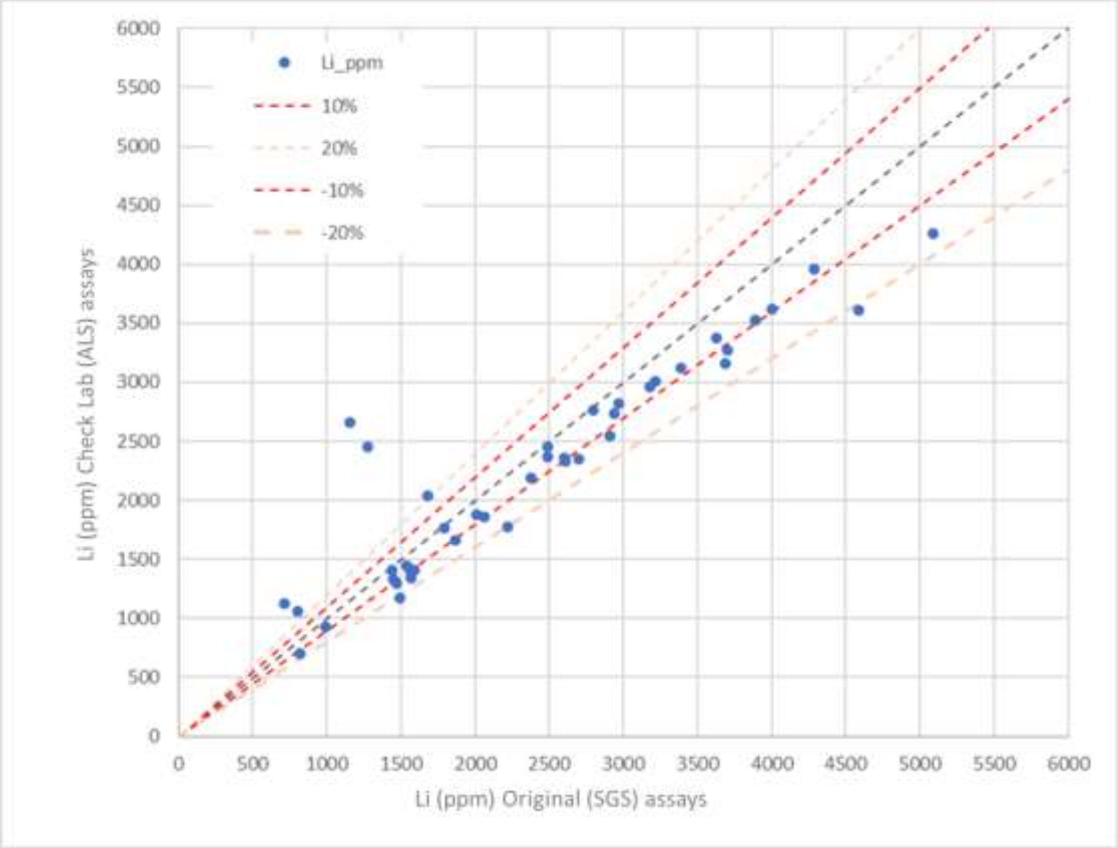
Summary of analytical methods used and detection limits for 2017 exploration programme

Method code	Elements	Lower detection limit	Upper detection limit	Description
ME-MS89L	5 elements* (Fe, Li, Nb, Sn, Ta)	2 ppm Li	25,000 ppm Li	Multi-Element lowest detection limit method using Na_2O_2 fusion and ICP-MS. Na_2O_2 Fusion followed by ICP-MS. Method Precision: ± 10–15 %.

Note: ** Detection limit varies depending on element*

Figure 11-7

Plot of original samples vs check samples (n=41)



The results from the check laboratory (ALS-Chemex) reported lower Li values compared to the original (SGS) analyses with a positive 8 % bias to the SGS results, with the two outliers are removed. When the average results for AMIS338 are compared to those reported by SGS, the same bias is observed for SGS versus ALS (Figure 11-4A and Figure 11-8). This bias can be attributed to slight differences in the preparation and analytical methods between the laboratories.

Table 11-5

Table showing analytical bias between primary and check laboratories using AMIS0338

AMIS0338	Number of samples	Average (ppm)	% difference
SGS	28	1818	+6
ALS	6	1690	-2
Certified Value	-	1742	-

The AMIS0338 check samples reported acceptable values, close to the expected certified values (Figure 11-8). The check samples of AMIS0484 report Li values between 7 and 19 ppm (Figure 11-9).

Figure 11-8

Plot of AMIS0338 check results (n=6) from the current exploration programme

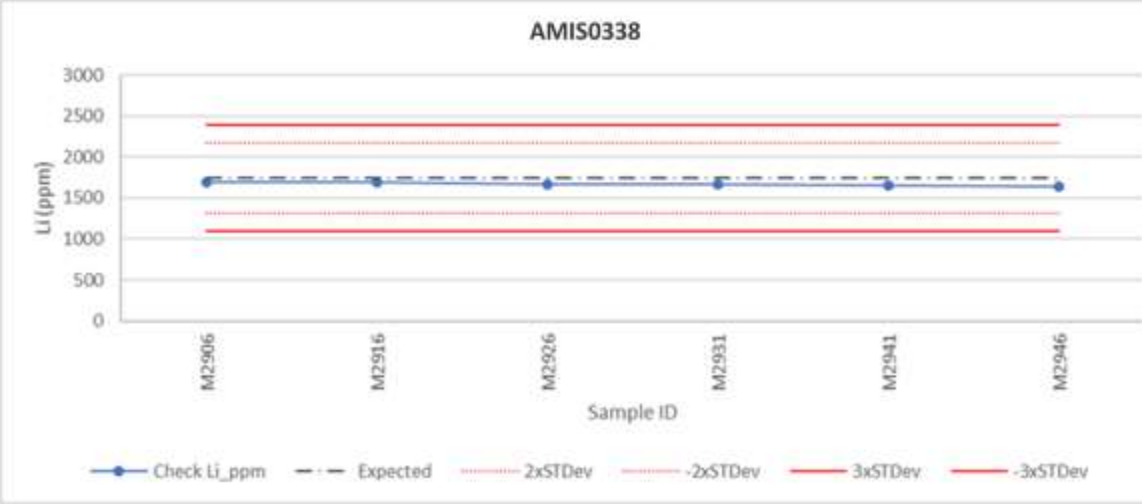
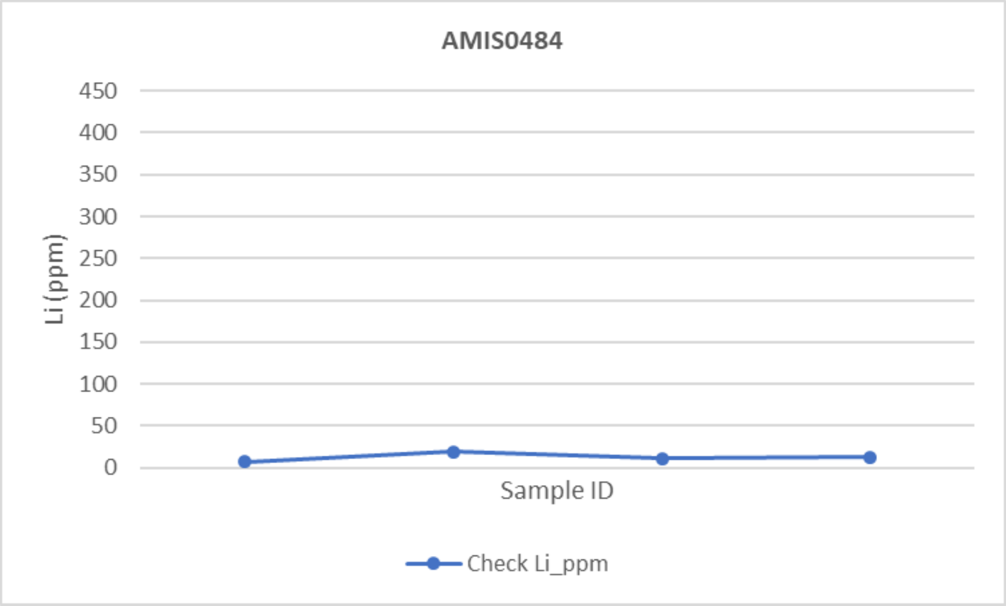


Figure 11-9
Performance of AMIS0484 check results (n=4) from the current exploration programme



11.3 Conclusion

The QP is of the opinion that the sample handling and logging protocols are considered in line with industry practice and that the samples are representative of the mineralization. Overall the performance of the CRMs, blanks, duplicates and check laboratory samples are considered acceptable. The assay methods used by SGS are in line with accepted industry standards and the assay results considered acceptable for use in the Mineral Resource estimate.

12 DATA VERIFICATION

The data verification included the following:

- a review of the historical data and Lintmar reports by MSA; and
- a site visit by Mr Cronwright and check sampling and assay of the Kamativi tailings in 2017.
- a site visit from 23-24 April 2018 by Mr Cronwright during the execution of the 2018 drilling campaign.

12.1 Verification of Previous Sampling Campaign

Verification sampling was conducted by MSA during the site visit by the QP, Mr Cronwright, on 7 August 2017. Ten samples were taken from locations previously sampled by Lintmar in 2016 and 2017 and a number of locations previously not sampled. Two AMIS0343 CRM samples sourced from AMIS, and two AMIS0439 blanks comprising silica chips, were also inserted into the sample batch.

All the samples were collected by the QP and comprised samples taken using a spade from either the sides of the tailings dump or pre-existing pits. Two samples, a primary and a check sample, of between 1-3 kg each, were collected from each location and inserted into plastic bags along with a pre-numbered sample tag. The bags were sealed with a numbered tamper proof cable tie. The bags were also labelled with a permanent marker using the same number as that of the sample tag. Photographs were taken of all the sample tags and a hardcopy record kept of all samples collected.

The samples were then transported by Lintmar to MSA's offices in Johannesburg and the QA/QC samples inserted into the sample batch on 18 August 2017. The primary samples were then submitted to SGS in Johannesburg on 19 August 2017 and the results reported on 25 August 2017. All samples were assayed as per Table 11-1.

The QA/QC sample results from the verification sampling are considered acceptable and summarised as follows:

- both of the blank samples, AMIS0439, reported Li values <4x the lower detection limit (i.e. <40 ppm Li); and
- the two AMIS0343 samples reported Li values lower than the certified Li value of 7150 ppm Li, but within 10 %.

Comparison of the results from the MSA check sampling (L4646 to L4656) with the results of the sampling previously conducted by Lintmar shows a reasonably good correlation considering the inherent non-representivity of sampling (Figure 9-2). It is noted that the Li results of the check sampling are on average 20 % lower than the results from the original samples but are within a similar range of Li-values and are thus considered suitable for use in this report.

12.2 Verification of 2018 Exploration Data

All data generated during the 2018 drilling programme were collected by the MSA project geologist managing the project and data managed by MSA in accordance with the relevant standard operating procedures as summarised in sections 10 and 11, The MSA project geologist managing the project performed regular checks on the surveyor data to ensure the collar locations were recorded correctly.

All field data were captured into a series of Microsoft Excel spreadsheets and validated by MSA's data management department. The data were checked for duplicate sample entries, missing samples, sample overlaps. Any queries that were raised were reverted to the project geologist who took appropriate action.

During the QP's site visit from 23-24 April 2018 the locations of the data points were verified and adherence to the SOPs for the logging, density pitting and sampling checked. No issues regarding non-adherence to the SOPs were noted.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

A campaign of preliminary metallurgical test work was conducted in two phases during 2015 and 2016. Test work assessed the deportment of lithium, tin, tantalum and iron, and investigated the potential recovery and concentration of these elements. The work was based on the following:

- mineralogical characterisation;
- heavy liquid separation ("HLS") tests using tetrabromoethane ("TBE") as a dense medium;
- X-Ray Fluorescence ("XRF") analysis of the head feed and X-ray powder diffraction ("XRD") analysis of the HLS floats and sinks fractions;
- Quantitative Evaluation of Minerals by Scanning Electron Microscopy ("QEMSCAN"), bulk modal mineralogy and mineral liberation analyses on the HLS sinks fraction; and
- magnetic separation.

13.1 Sampling

Two sampling campaigns for mineral processing and metallurgical test work have been conducted on the Kamativi dumps. During April 2015 samples KS1 and KS2 were taken from historical sampling pits and trenches (see Section 11 for details on the sample procedures). These samples together with a second batch of samples (samples Li-A to Li-G and KM1) taken during June 2016 and July 2016 (see Section 11 for details on the sample procedures) were submitted to SGS South Africa and form the basis of the assays and test results recorded in this section of the report.

Samples KS-1 and KS-2 were excavated as large samples of approximately 100 kilograms each and reduced to smaller sub-samples by the "cone-and-quarter" technique to obtain representative sub-samples of approximately 25 kilograms each.

In June 2016, additional bulk samples were excavated from various areas of the dump. Individual sub-samples were again produced by cone-and-quartering larger material masses excavated from the dump. Seven separate samples were delivered to South Africa for test work. These samples were designated LI-A to LI-G. The rejects from these samples were combined and split into two separate KM1 and the AH series of samples sub-samples, which were submitted for additional test work.

A summary of the completed and outstanding test work by sample is presented in Table 13-1 and section 13.8.

Table 13-1
Summary of metallurgical samples and test work

Sample Designation	Test work Description
KS-1, KS-2	Elemental analysis (ICP-OES/MS), grade by size fraction, particle size distribution (PSD) analysis
LI-A to G (Elemental Analysis) KM1 (Mineralogy)	Elemental Analysis (ICP OES/MS), mineralogy. The rejects from these samples were combined and split into KM1 and the AH-series of samples.

13.2 Test Work and Results

13.2.1 Elemental and Particle Size Distribution Analyses

Elemental analysis was completed by means of inductively coupled plasma ("ICP") technology. The head elemental analysis is shown for Li_2O, iron, tin and tantalum in Figure 13-1 to Figure 13-4. The head samples contain an average of 0.67 % Li_2O, 274 ppm tin, 17 ppm tantalum and 0.90 % iron. The Li_2O, iron and tantalum grades are fairly consistent, increasing confidence that the grades of these elements, with the exception of tin, do not vary significantly across the surface of the dump. The tin grade is much more variable, as shown in Figure 13-3.

Figure 13-1
Li_2O head feed grade of metallurgical test work samples

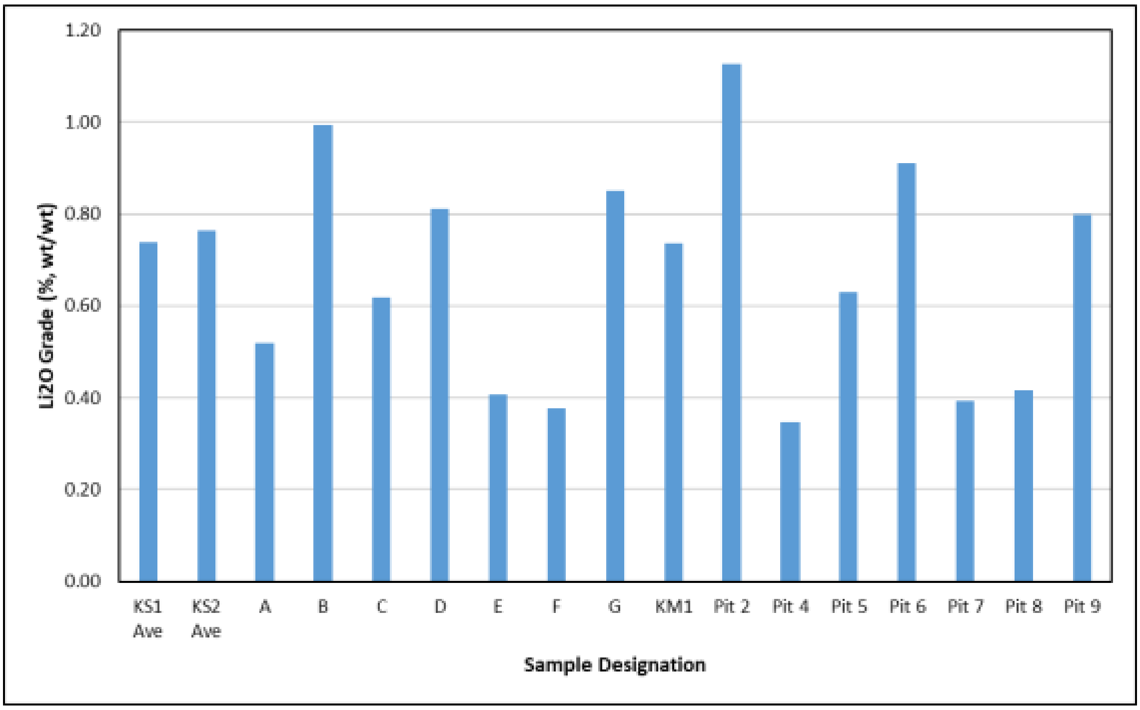


Figure 13-2
Iron head feed grade of metallurgical test work samples

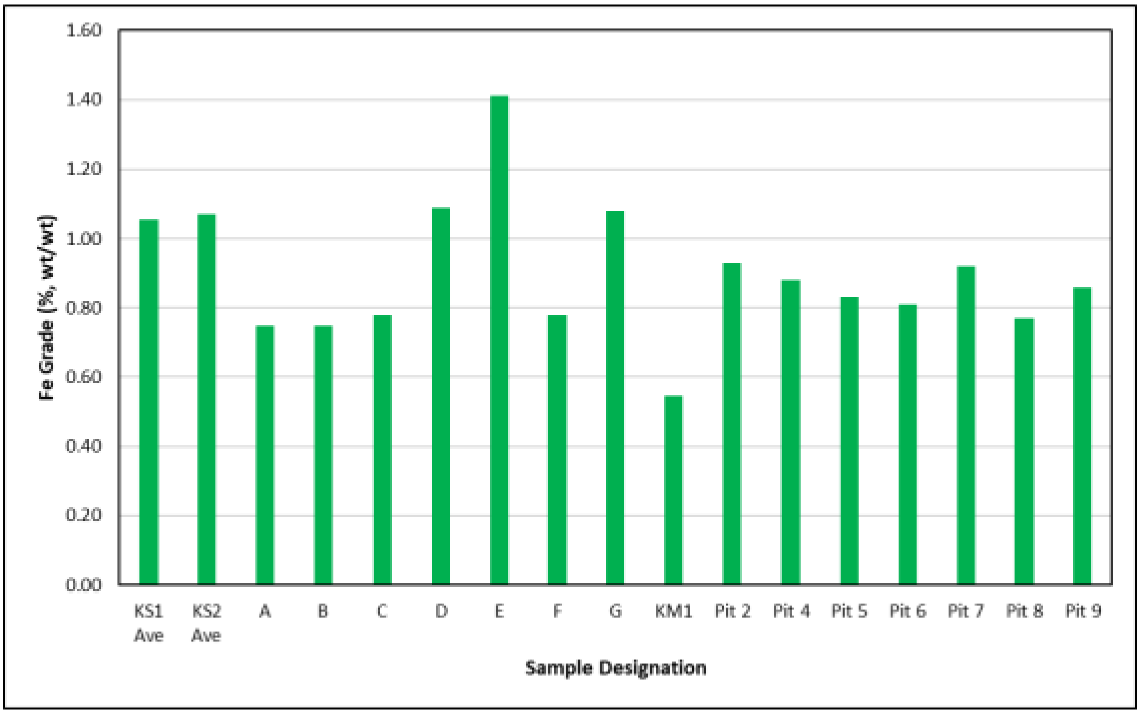


Figure 13-3
Tin head feed grade of metallurgical test work samples

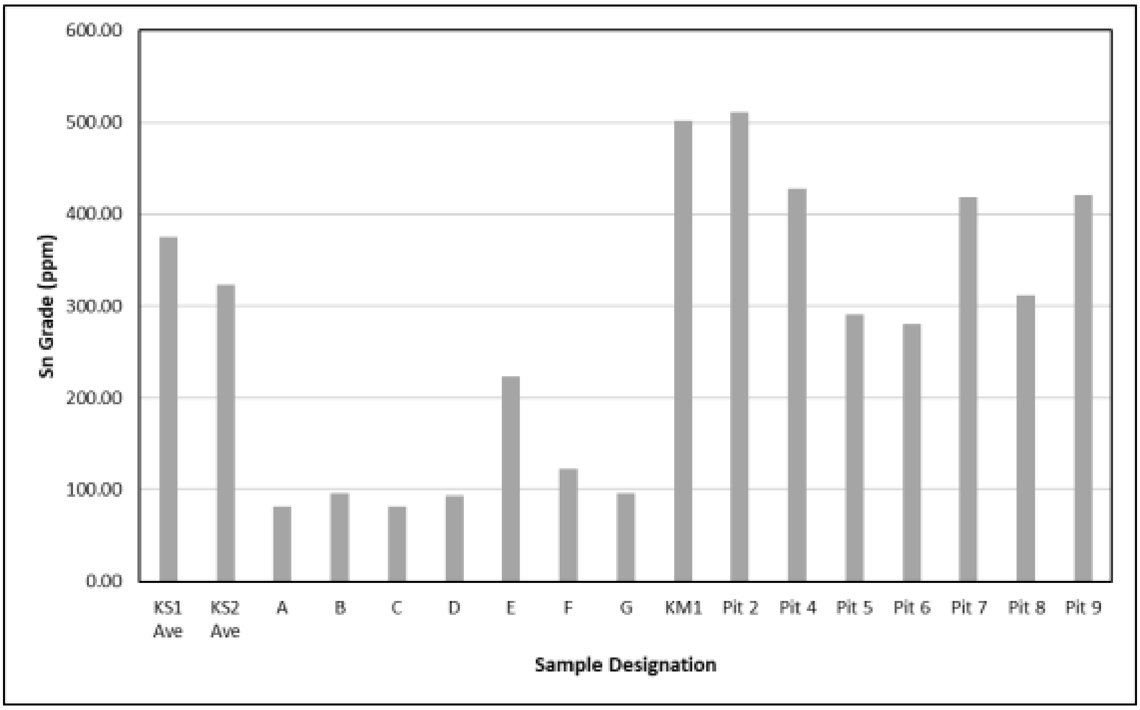
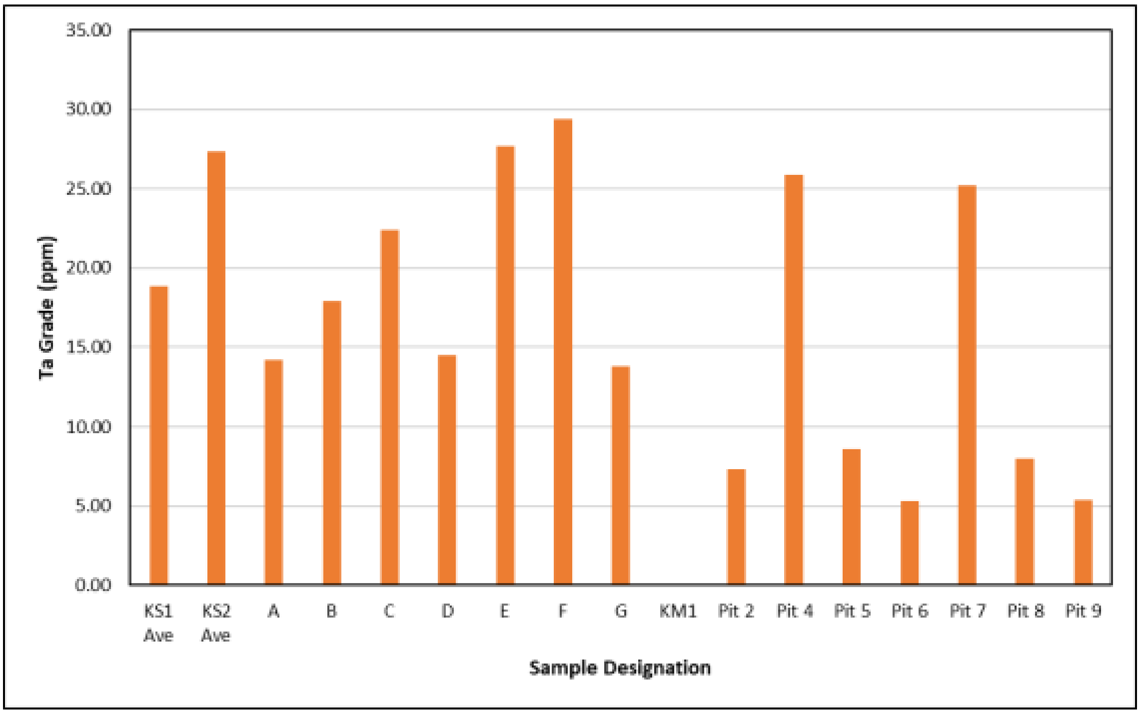


Figure 13-4
Tantalum head feed grade of metallurgical test work samples



Analysis of the samples taken in 2015 (KS1 and KS2) was also conducted on the various size fractions. Li_2O, iron, tin and tantalum grades within each size fraction are shown for samples KS1 and KS2 in Figure 13-5 to Figure 13-8.

Figure 13-5 indicates that the lithium-bearing minerals are relatively uniform across the entire size range, although slightly depleted in the finer fractions. Tin and tantalum, shown in Figure 13-7 and Figure 13-8, are concentrated in the finer size fractions (-75 µm). Iron is concentrated within the 106 µm to 53 µm size range, as illustrated in Figure 13-6. This distribution of elements raises the possibility that the iron content of the feed to flotation can be reduced significantly by de-sliming the material. The tin and tantalum will also be concentrated in the slimes, allowing for possible beneficiation of this fraction for these elements. It is likely however, that the tantalum grade will prove too low for economic extraction. The extraction of tin may possibly be achieved by the application of fine material gravity separation technology (reflux classifier). This possibility will be investigated as test work progresses. The de-sliming of the flotation plant feed is also beneficial for the flotation process itself, as the presence of slimes and micaceous gangue material tends to result in entrainment and the formation of a more viscous froth, which in turn results in decreased flotation efficiency and product grade.

Figure 13-5

Li_2O head feed grade by size analysis

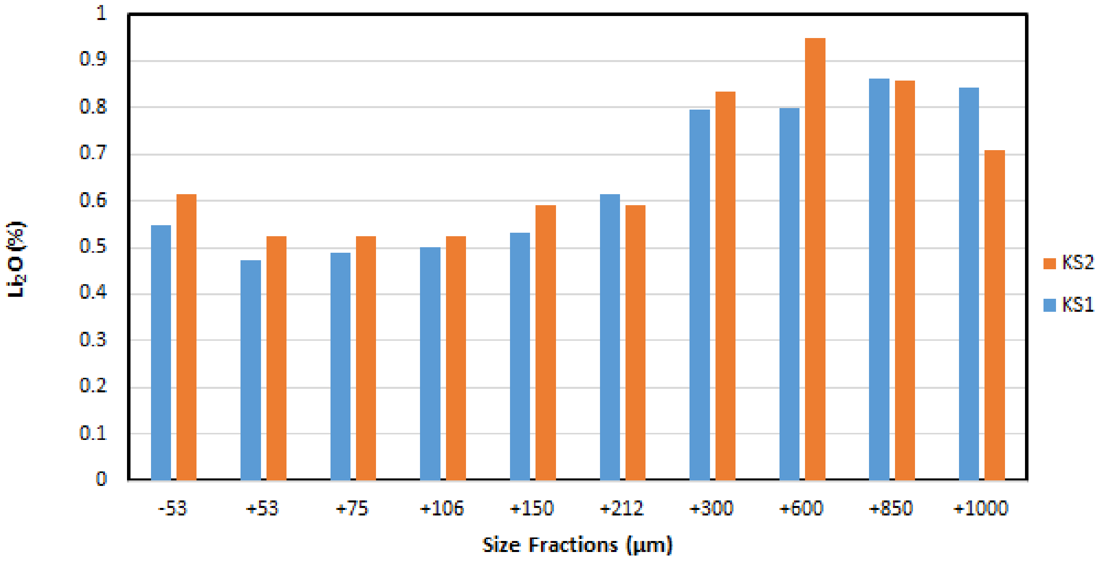


Figure 13-6

Iron head feed grade by size analysis

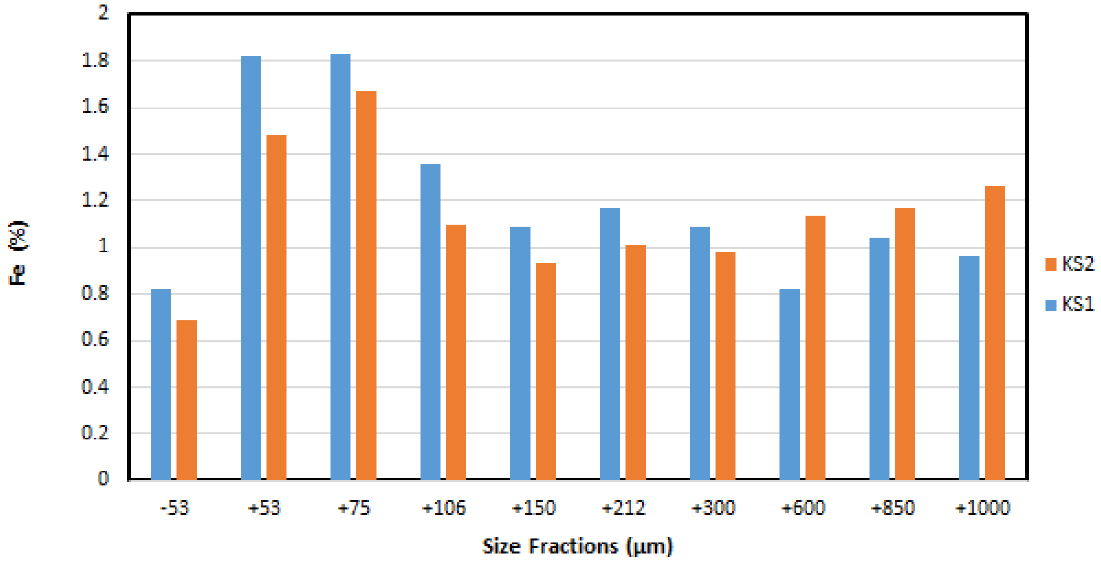


Figure 13-7
Tin head feed grade by size analysis

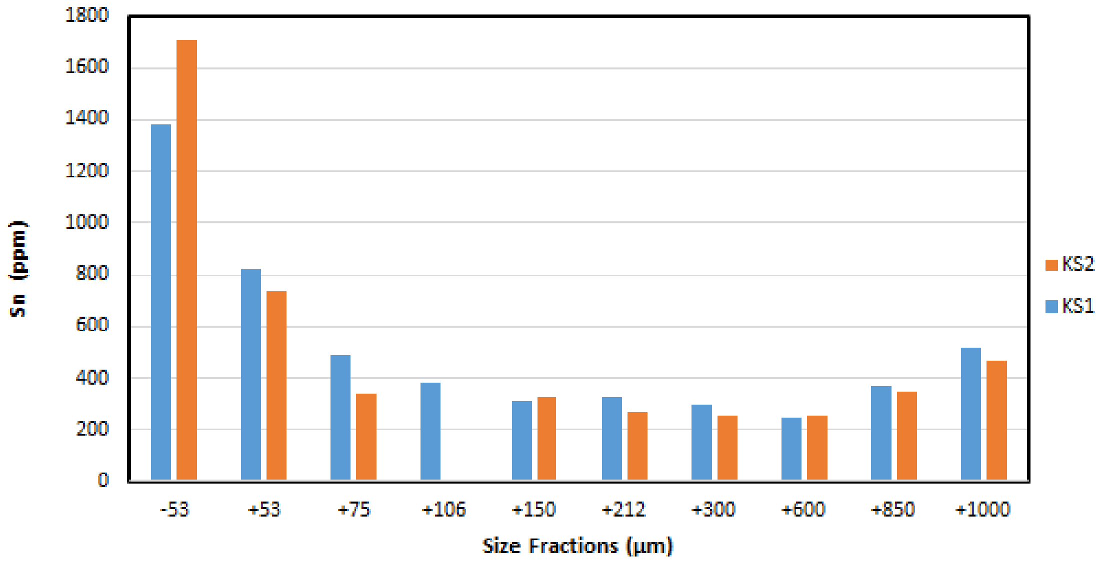
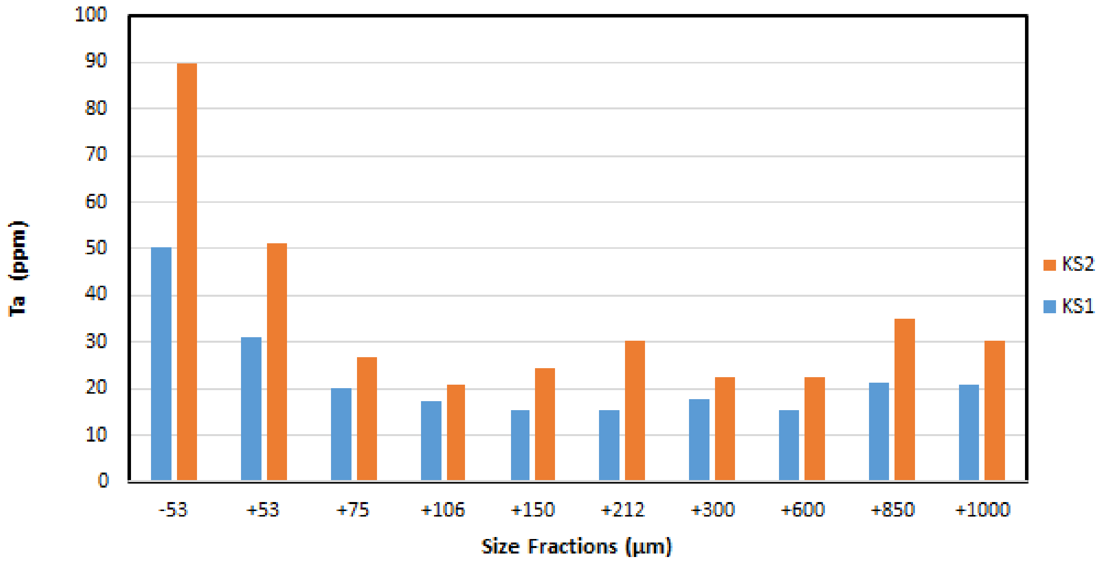


Figure 13-8
Tantalum head feed grade by size analysis



Grade-by-size analysis was also completed on the 2016 mineralogy sample KM1. The resultant data is presented in Figure 13-9 and Figure 13-10. The concentration of iron and tin within the finer size fractions is confirmed by the data presented. The trend observed for Li_2O is similar to that associated with the 2016 data, samples KM1 and LI-A to G.

Figure 13-9

Mineralogy sample grade by size for Li_2O and Fe (iron)

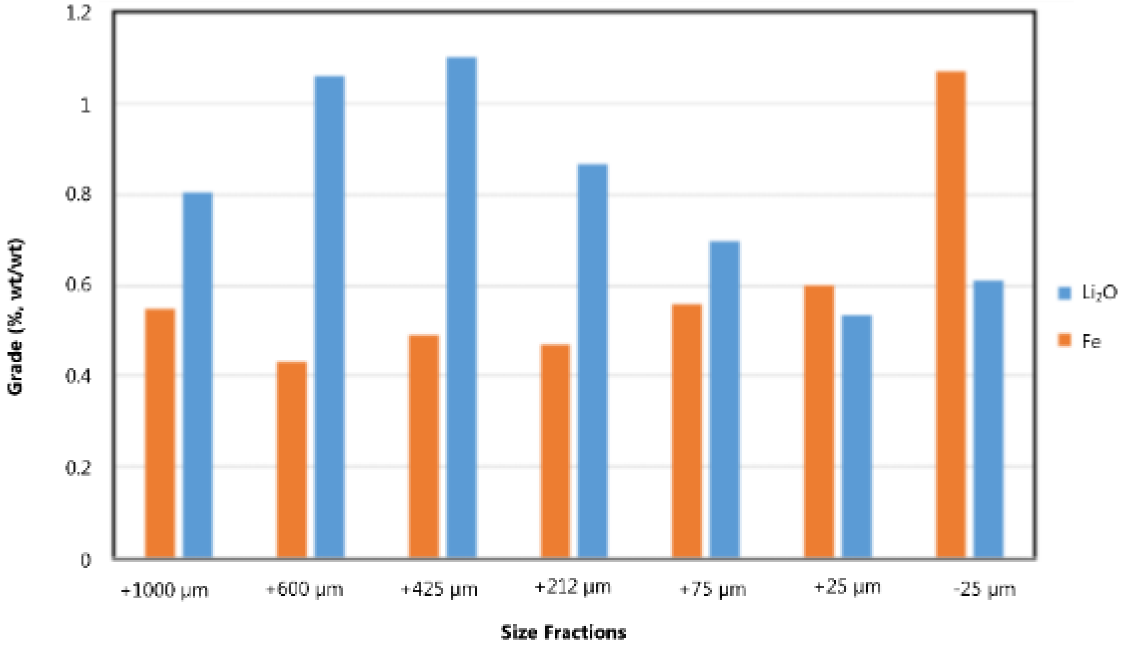
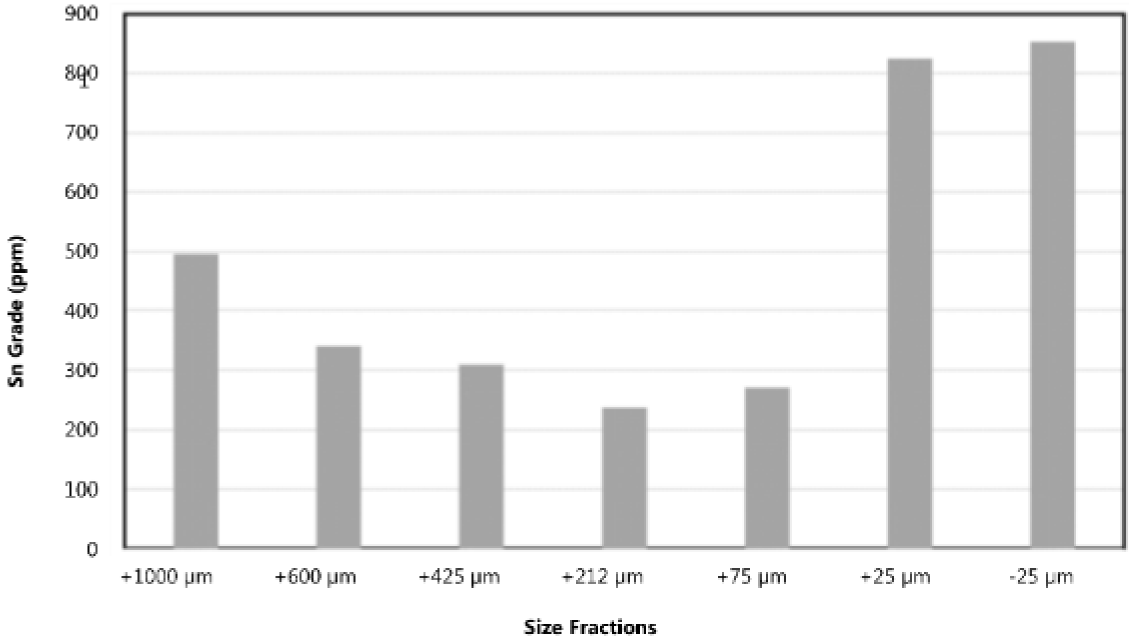


Figure 13-10

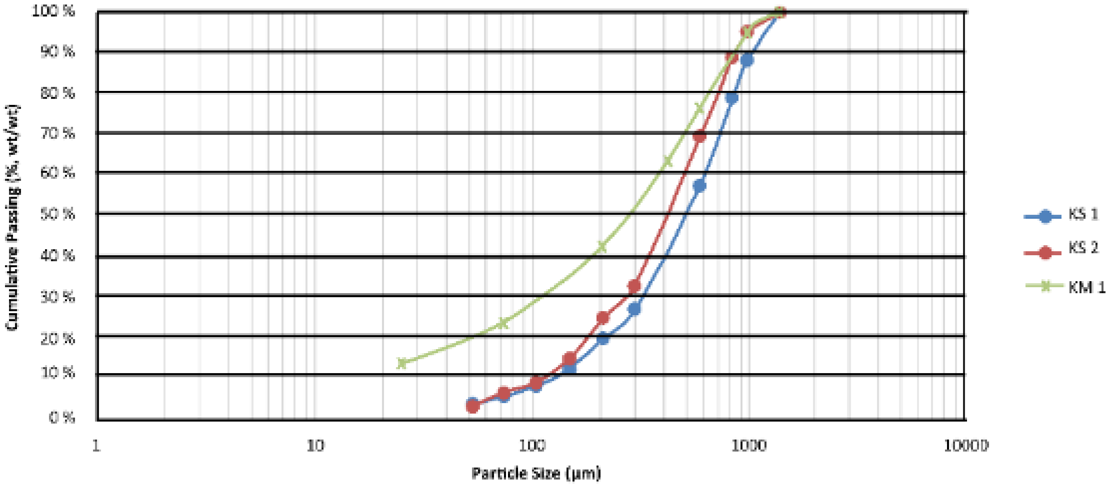
Mineralogy sample grade by size for Sn (tin)



Particle size distribution analysis was completed on the 2015 samples, KS-1 and KS-2, and on the mineralogy sample, KM-1. The particle size distribution for these individual samples is shown in Figure 13-11. From the data it is clear that the mineralogy sample (KM-1) is significantly finer than the 2015 samples. The d50 values for samples KS-1 and KS-2 are 500 µm and 440 µm respectively, while the d50 for sample KM-1 is 290 µm. This is likely due to the fact that KS-1 and KS-2 were taken from single sites on the sides of the dump, while sample KM-1 is a larger more representative composite sample excavated from a number of points across the surface area of the dump. The material deposited towards the centre of the dump is generally finer.

Figure 13-11

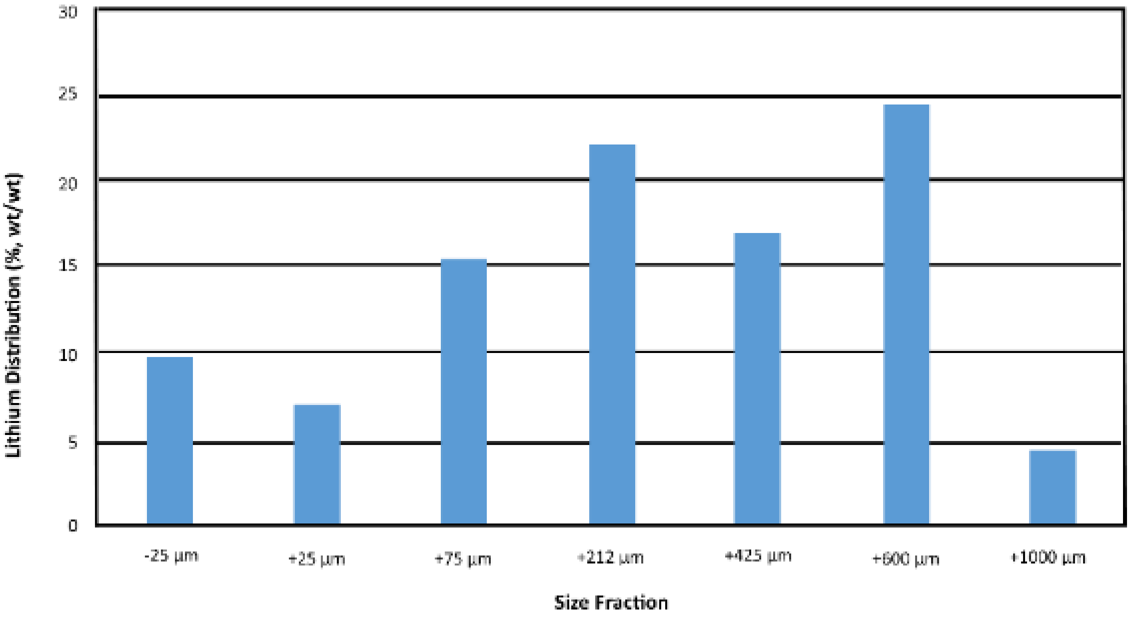
Particle size distribution of head samples



The distribution of lithium across the various size ranges was calculated from analysis of particle size and grade-by-size distribution data. The distribution of the element is presented in Figure 13-12. The majority of the lithium is contained within the 75 µm to 1,000 µm size fraction.

Figure 13-12

Distribution of lithium across the various size fractions for the mineralogy sample KM-1



13.2.2 Mineralogical Characteristics

Mineralogical work was carried out by SGS Mineral Services, a respected and prominent laboratory based in Randfontein, South Africa. Test work conducted at SGS included:

- XRF analysis of the head feed;
- HLS tests using TBE as a dense medium;
- XRD analysis of the HLS floats and sinks fractions; and
- QEMSCAN, bulk modal mineralogy and mineral liberation analyses on the HLS sinks fraction.

13.2.2.1 Head feed XRF and HLS fraction bulk modal compositions

The head feed XRF results are presented in Table 13-2. The silica and aluminium values indicate that most of the material is present as aluminium silicates and quartz. The K_2O, Na_2O and CaO values indicate that a significant amount of the material is comprised of various feldspar minerals. The Fe_2O_3 grade of 0.86 % translates to an elemental iron grade of 0.6 % Fe. This differs slightly from the ICP value of 0.55 % Fe which is attributed to experimental error associated with the XRF measurement. The iron is present in the form of iron oxides and also in other minerals such as garnet, iron phosphates and tourmaline.

Table 13-2
Head Feed XRF Analysis

Compound	Al_2O_3	SiO_2	Fe_2O_3	MgO	CaO	K_2O	Na_2O
Grade (% wt/wt)	16.8	71.6	0.86	0.07	0.59	2.58	4.61

The results from the HLS test and the mass distribution of lithium between the different HLS fractions is shown in Table 13-3. A total of 5.6 % of the head sample mass reports to HLS sinks, while 81.4 % reports to the floats and 13 % is considered as slimes material (-25 µm). The Li_2O grade of the sinks material is 6.68 %, and accounts for approximately 50 % of the lithium mass in the sample.

Table 13-3
Heavy liquid separation results

HLS Fraction	Mass (g)	Mass Distribution (% wt/wt)	Li Grade (ppm)	Li_2O Grade (% wt/wt)	Li_2O Distribution (% wt/wt)
Sinks	29.11	5.59	31,000	6.68	49.65
Floats	423.46	81.36	1,720	0.37	40.08
Slimes	67.88	13.04	2,750	0.59	10.27
Total	**520.45**	**100.00**	**3,492**	**0.75**	**100.00**

The distribution of various minerals within the HLS fractions (Bulk Modal Analysis) is shown in Table 13-4. The presence of feldspar and quartz is confirmed by the Bulk Modal Analysis. Significant amounts of quartz, plagioclase (sodium and calcium feldspars) and K-feldspar are found in the HLS floats and slimes fractions. The Bulk Modal Analysis indicates that approximately 80 % of the sinks fraction mass comprises spodumene. A significant amount of lithium bearing material can be found in the floats fraction and after analysis of the data presented in Table 13-3 and Table 13-4, it is clear that a significant portion of this material is in the form of occluded spodumene particles. Smaller amounts of petalite, which is less dense than TBE, are also present. Trace amounts of lithium may also be in the form of amblygonite, which would report to the sinks fraction.

Table 13-4
Bulk Modal Compositions

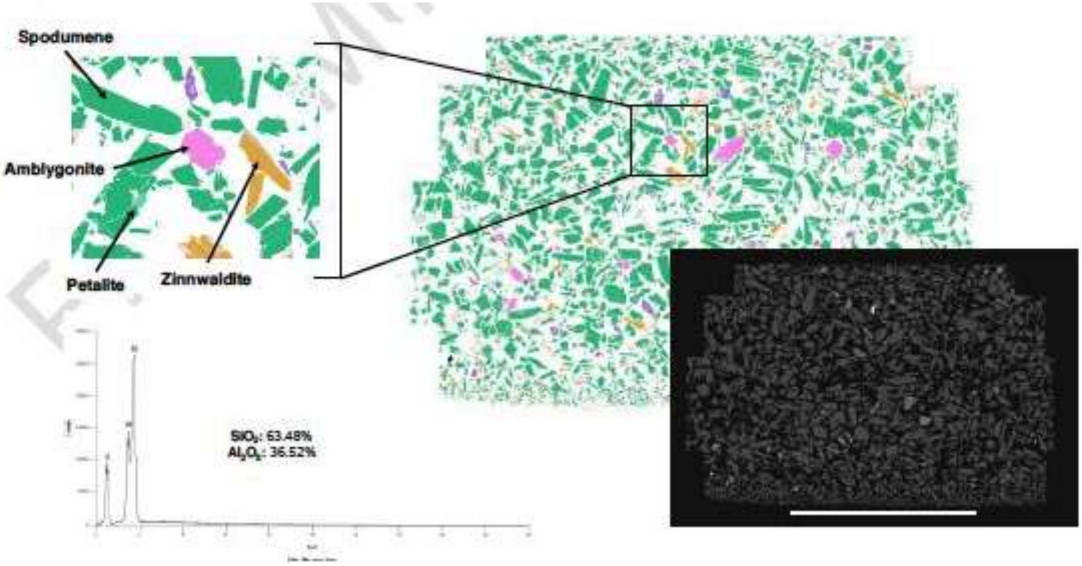
Mineral	Calculated Head Grade (% wt/wt)	HLS Sinks Grade (% wt/wt)	HLS Floats Grade (% wt/wt)	Slimes Grade (% wt/wt)
Plagioclase	34.74	1.09	38.74	24.19
Quartz	30.31	3.54	34.20	17.52
Muscovite	12.05	1.53	10.99	23.16
K-feldspar	7.93	0.11	8.42	8.24
Tourmaline	0.54	1.50	0.35	1.35
Kalborsite	0.21	0.76	0.15	0.34
Dumortierite	0.18	0.08	0.10	0.76
Other Silicates	0.06	0.57	0.04	0.02
Total Non-Lithium Containing Silicates	**86.07**	**9.21**	**93.03**	**75.63**
Spodumene	6.44	80.84	1.83	3.27
Cookeite	3.19	0.97	1.66	13.70
Zinnwaldite	1.39	2.67	1.39	0.85
Petalite	1.12	0.69	0.87	2.83
Amblygonite	0.31	1.30	0.20	0.54
Total Lithium Bearing Minerals	**12.44**	**86.47**	**5.95**	**21.18**
Cassiterite	0.03	0.23	0	0.09
Fe(Mn) Oxides	0.23	1.04	0.14	0.40
Other Oxides	0.05	0.05	0.05	0.04
Total Oxide Minerals	**0.30**	**1.32**	**0.19**	**0.53**
Apatite	0.92	1.85	0.64	2.24
Other Phosphates	0.12	1.10	0.06	0.04
Other	0.16	0.05	0.13	0.37
Total Phosphates and Other Minerals	**1.20**	**3.00**	**0.83**	**2.65**

Lithium deportment data for the various streams is presented in Table 13-5 and Figure 13-13. Approximately 70 % of the lithium in the feed material is in the form of spodumene, while 96 % of the lithium contained within the sinks fractions is in the form of spodumene. The lithium deportment to spodumene within the floats fraction is approximately 50 %.

Table 13-5
Lithium deportment from HLS test work

HLS Fraction	Lithium Deportment (% wt/wt)				
	Spodumene	Cookeite	Zinnwaldite	Petalite	Amblygonite
Concentrate/Sinks	96.34	0.41	1.36	0.46	1.43
Floats	49.66	16.02	16.03	13.28	5.00
Slimes	30.88	46.08	3.40	14.97	4.66
Calculated Head	**70.95**	**12.53**	**6.52**	**6.91**	**3.10**

Figure 13-13
QEMSCAN and BSE maps of HLS concentrate polished thin section. EDS spectra from a spodumene grain. Scale bar = 11 mm



Analysis was also conducted on the sinks material to determine iron deportment to the various iron-bearing mineral species. Data relating to iron deportment is presented in Table 13-6. The majority of the iron in the sinks fraction is present in the form of iron manganese oxides, zinnwaldite, garnet, tourmaline and iron phosphates.

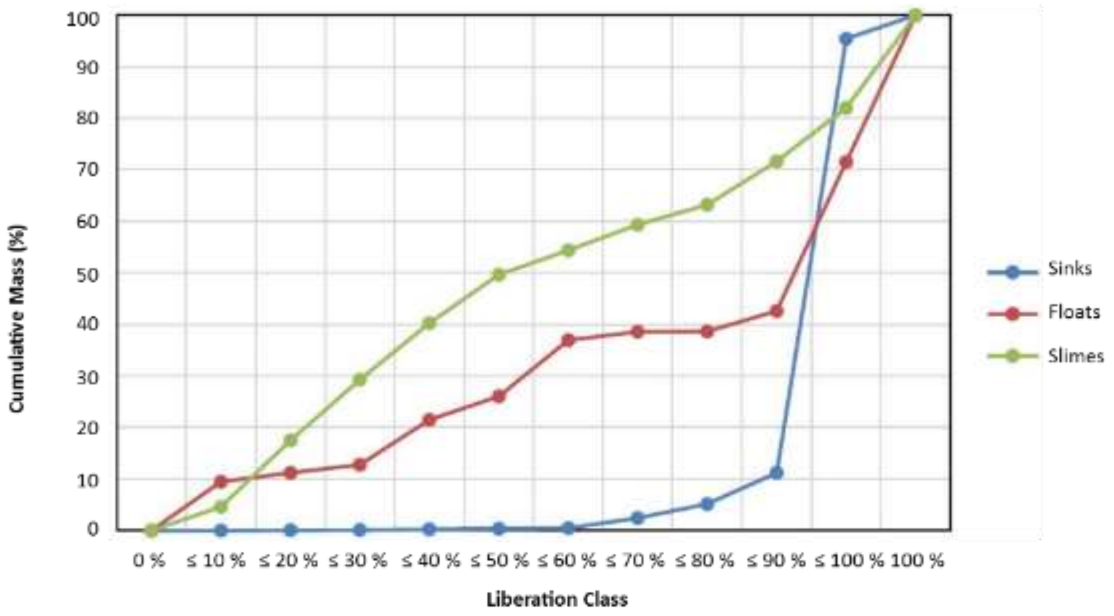
Table 13-6
Iron deportment

Mineral Species	Iron Deportment (%)
Fe(Mn) Oxides	34.67
Zinnwaldite	21.62
Garnet	15.43
Iron Phosphates	15.25
Tourmaline	10.12
Muscovite	1.39
Other	1.51

13.2.2.2 Particle liberation

The completion of QEMSCAN analyses allowed for a detailed examination of the liberation characteristics of various minerals contained within the HLS material fractions. Liberation data for spodumene is presented in Figure 13-14.

Figure 13-14
Spodumene liberation



It is concluded that:

- spodumene is well liberated in the HLS sinks fraction where in the order of 95 % of the particles are more than 80 % liberated;
- similarly, a relatively large component is also well liberated in the HLS floats fraction, with approximately 61 % of the spodumene particles characterised by liberation in excess of 80 %; and
- the liberation of spodumene in the slimes fraction is poor. However, this fraction accounts for a relatively small percentage of the total lithium content.

Further QEMSCAN and Back Scatter Electron ("BSE") work (Figure 13-3) conducted by SGS on the HLS fraction also confirmed that the spodumene has high levels of liberation. Most of the lithium bearing minerals are also present, to a lesser extent, within the sinks fraction. Some phosphates and quartz species are present, which constitutes the diluents within the HLS sink fraction.

The QEMSCAN analysis also offers more detailed insight into the association of the spodumene particles with other minerals within the various HLS fractions. Table 13-7 represents the mineral association data for spodumene for the sinks, floats and slimes fractions.

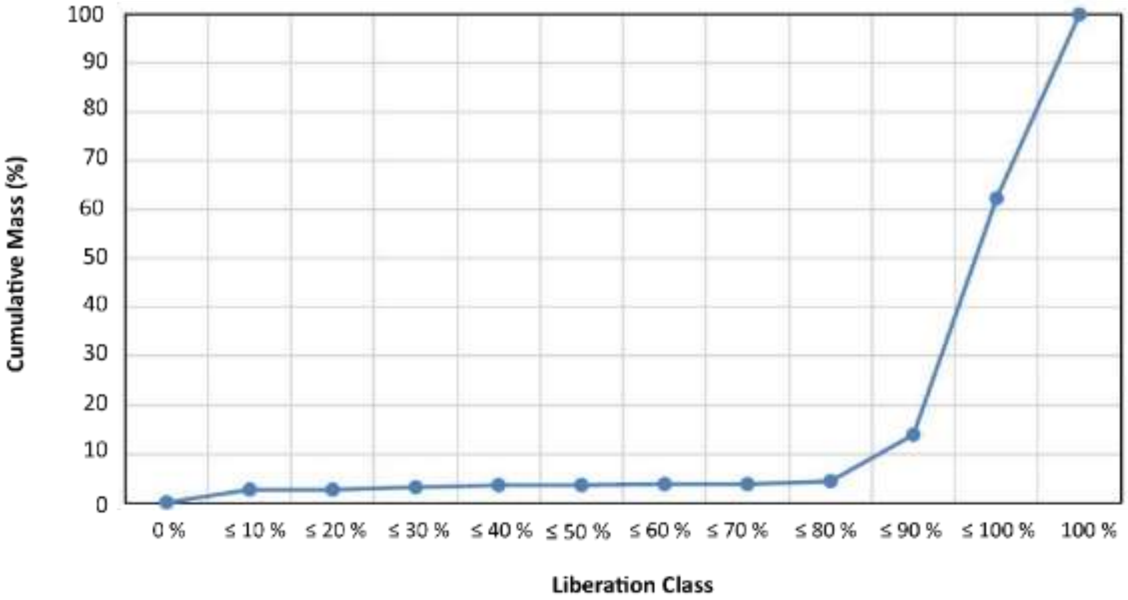
Table 13-7
Mineral association of Spodumene

HLS Fraction	Sinks	Floats	Slimes
Mineral	**Association with Spodumene (% wt/wt)**		
Background	78.21	58.60	51.62
Plagioclase	3.75	7.97	8.43
Quartz	5.84	12.61	4.25
Muscovite	3.53	4.83	16.31
Cookeite	3.69	7.44	12.66
Petalite	3.34	7.05	4.07
Other	1.64	1.50	2.66

Within the sinks fraction 78.2 % of the spodumene crystals are fully liberated. Attached spodumene within the sinks fraction is predominantly associated with plagioclase, quartz, muscovite, cookeite and petalite. The poorer liberation of the spodumene within the floats and slimes fraction is evidenced by the lower background associations of 58.60 % and 51.62 % for the spodumene in the floats and slimes respectively.

QEMSCAN data indicates that all the tin in the material occurs as cassiterite (SnO_2). Mineral liberation data relating to this mineral is presented in Figure 13-15, for the sinks material fraction only. The data indicates that the cassiterite grains within this material class are very well liberated with approximately 96 % of the cassiterite exhibiting liberation in excess of 80 % by particle area.

Figure 13-15
Cassiterite liberation – sinks fraction



13.3 Preliminary Test Work Summary

The preliminary test work indicates that:

- the head samples contain an average of 0.74 % Li_2O, 316 ppm Sn, 24 ppm Ta and 0.83 % Fe. The individual sample Li_2O, iron and tantalum grades are fairly consistent, increasing confidence that the grades of these elements do not vary significantly across the surface of the dump. The tin grade is much more variable;
- mineralogical analysis indicates that the head material contains approximately 6.4 % spodumene, and that a concentrate containing 96.3 % spodumene (6.68 % Li_2O) can be produced by means of HLS separation at an SG of 2.96. Of the feed material, approximately 5.6 % reports to the HLS sinks fraction. The spodumene within the sinks fraction is well liberated, with 78.2 % of the crystals being entirely liberated. Attached spodumene within the sinks fraction is predominantly associated with plagioclase, quartz, muscovite, cookeite and petalite. The poorer liberation of the spodumene within the floats and slimes fraction is evidenced by the lower background associations of 58.60 % and 51.62 % for the spodumene in the floats and slimes respectively.

13.4 Potential By-Products and Deleterious Elements

During the course of the preliminary test work, consideration has been given to potential commercial by-products and deleterious elements likely to impact on the saleability of a Li_2O concentrate product.

Tin may potentially be a useful by-product, while iron is a penalty element and data relating to its content in a potential HLS or flotation concentrate should be closely analysed. Cassiterite is concentrated in the HLS sinks and slimes fractions. 37.2 % of the cassiterite reports to the slimes (-25 µm) fraction. The deportation of a large proportion of the cassiterite to the sinks fraction is due to the high density of the mineral. A significant proportion of the tin-bearing mineral may be recovered by the gravity concentration of the slimes fraction. The separation of slimes from the feed material may be achieved by means of a de-sliming cyclone.

A Mozley multi gravity separator ("MGS") unit is suitable for the treatment of the resultant cassiterite fines and is most efficient below 300 µm. The MGS utilises a similar working procedure to the shaking table with an additional centrifugal force applied to enhance the separation of fine particles. As the MGS is a low capacity unit, it is typically employed for the cleaning of precious metals or valuable minerals such as cassiterite from pre-concentrates. Further work will be required during the next phase of the Project.

Iron is generally specified as a penalty element in off-take agreements, with maximum iron grades relating to saleable spodumene concentrate being in the range of 0.1 - 0.2 % Fe. In light of the current test work the iron-bearing minerals would require removal from the HLS concentrate. This may be achieved through the use of magnetic separation equipment such as wet high intensity magnetic separation ("WHIMS"), possibly used in conjunction with gravity separation technology such as tabling. A single magnetic separation test has been performed which removed around 5 % of the total concentrate and produced a magnetic concentrate consisting of largely iron mineralisation. This will be further addressed during the next phase of the Project.

The Li2O – iron selectivity data indicates that the iron does not directly follow the spodumene recovery with an iron recovery of 53 % at a Li2O recovery of 71 %. The data suggests that an iron-rich mineral is reporting to the float along with the lithium-bearing minerals. The flotation of these iron species may be inhibited by the selection of an additional depressant and will be investigated further during the next phase of the Project.

13.5 Heavy Liquid Separation of Coarse Fraction

HLS test work was conducted by SGS during the first two weeks of June 2017. The purpose of this work was to evaluate the technical feasibility of the production of a spodumene pre-concentrate through the application of dense media separation (DMS) technology.

A concentrate was constituted from the HLS test work products for further magnetic separation test work, the results of which are discussed below.

13.5.1 Feed Material Preparation

A number of samples were taken from various pit locations across the tailings resource. Upon receipt of the material in South Africa, each of the samples were submitted to SGS for assay. Making use of the assay data provided by SGS, Cronimet Mining Processing South Africa (CMPSA) combined specified masses from the various samples to yield a 50 kg sample with a Li_2O grade of 0.75%. The material was subsequently submitted to SGS for the HLS test work.

Following sample receipt, SGS screened the sample at 425 µm. The HLS test work was to be conducted on the coarser fraction, while the -425 µm material would be stored by the laboratory for collection by CMPSA. Research on current spodumene extraction processes indicated that DMS is generally completed on a +500 µm material fraction for the production of a high Li_2O grade pre-concentrate, while the finer material (along with the DMS waste) is processed by means of flotation technology, for the recovery of further lithium bearing minerals.

The mass splits between the coarse and fine fractions are shown along with the relevant lithium (Li_2O) and tin grades, distributions in Table 13-8. Although a smaller mass proportion of the material is contained within the coarser material fraction (46.4 %), the Li_2O distribution within the +425 µm material fraction is higher, in the order of 55 % (m/m). This is due to the higher feed grade associated with this fraction, in the order of 0.85 % Li_2O.

Table 13-8

Coarse and fine fractions; mass split, Li_2O and Sn grades, distributions

	Mass (kg)	Li_2O Grade (%)	Li_2O Distribution (%)	Sn Grade (ppm)	Sn Distribution (%)	Mass Split (%)
Head Feed	47.2	0.714	100.00	316	100.0	100.0
+425 µm	21.9	0.853	55.10	307.0	40.7	46.4
-425 µm	25.3	0.601	44.90	387.0	59.3	53.6
Recalculated Head	-	0.72	-	349.91		-
Error on Head (%)	-	-0.52	-	-10.12		-

13.5.2 HLS Test Work

The coarse material fraction was subjected to HLS at various predefined specific gravities starting at an SG of 2.7 and concluding at an SG of 3.3 in incremental SG steps of 0.1. The dense media was constituted using mixtures of tetrabromoethane (TBE) and appropriate volumes of either acetone or ferrosilicon (FeSi).

The test work was conducted sequentially, commencing at an SG of 2.7. The sink fraction generated at an SG 2.7 medium was subjected to HLS at an SG 2.8 medium, with the sink fraction generated at each SG being collected and subjected to the subsequent and denser medium. The floats fraction at each SG was collected and stored for analysis. This methodology was followed until the sinks fraction was subjected to a medium density of 3.3.

The test work produced seven float fractions and one sink fraction. The sink and floats were washed with acetone, dried and weighed. A representative aliquot was split off, pulverised and submitted for Li and Sn by ICP.

Test work results are presented in Figures 13-16 and Figure 13-147. Mass recovery and grades, recovery of Li_2O, tin and iron are all considered. Tin may potentially be a useful by-product, while iron is a penalty element and data relating to its content in a potential DMS concentrate should be closely analysed. It should be noted that data in Figure 13-158to Figure 13-15 is plotted as a function of *separating SG or density* and therefore curves do not include the effect of the material generated in the floats fraction at a density of 2.7.

The Li_2O, tin and iron grades of each HLS sample taken during the course of the test work are shown in Figures 13-16 and 13-17. The lithium bearing species are concentrated within the heavier density ranges, much like tin, while the iron grades are more uniform across the SG range.

Figure 13-16

Li2O and Fe grades of HLS float samples

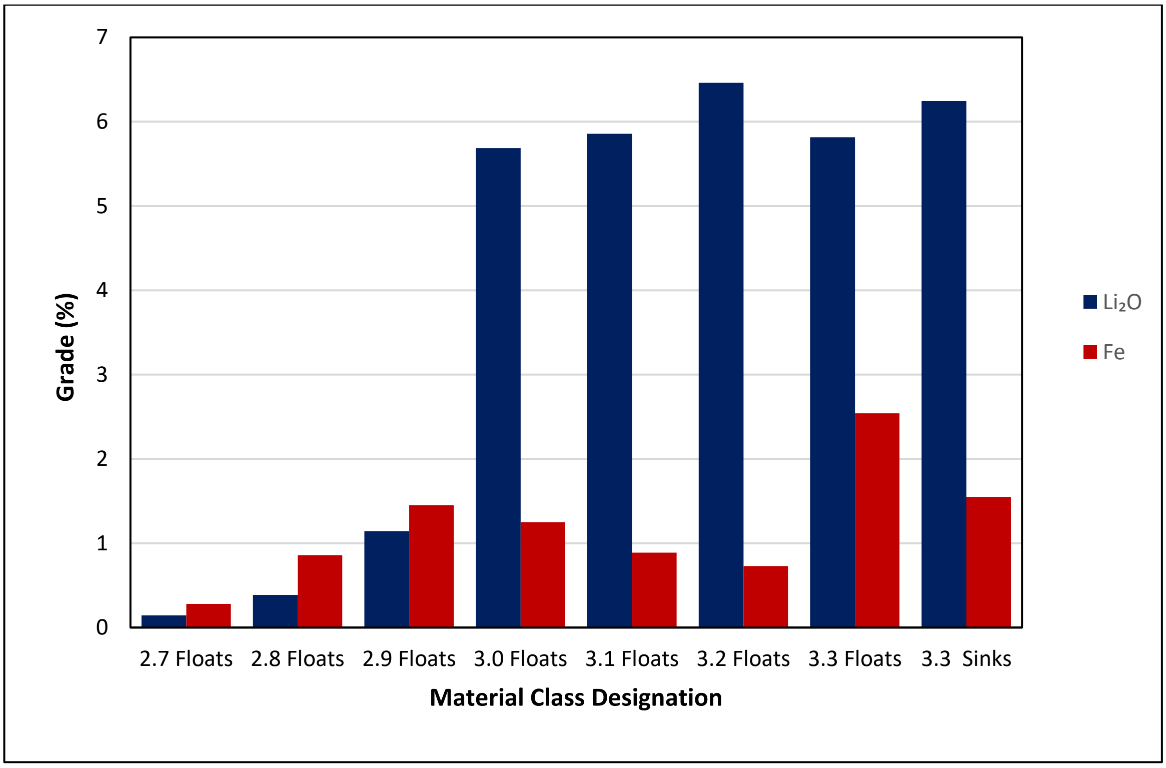
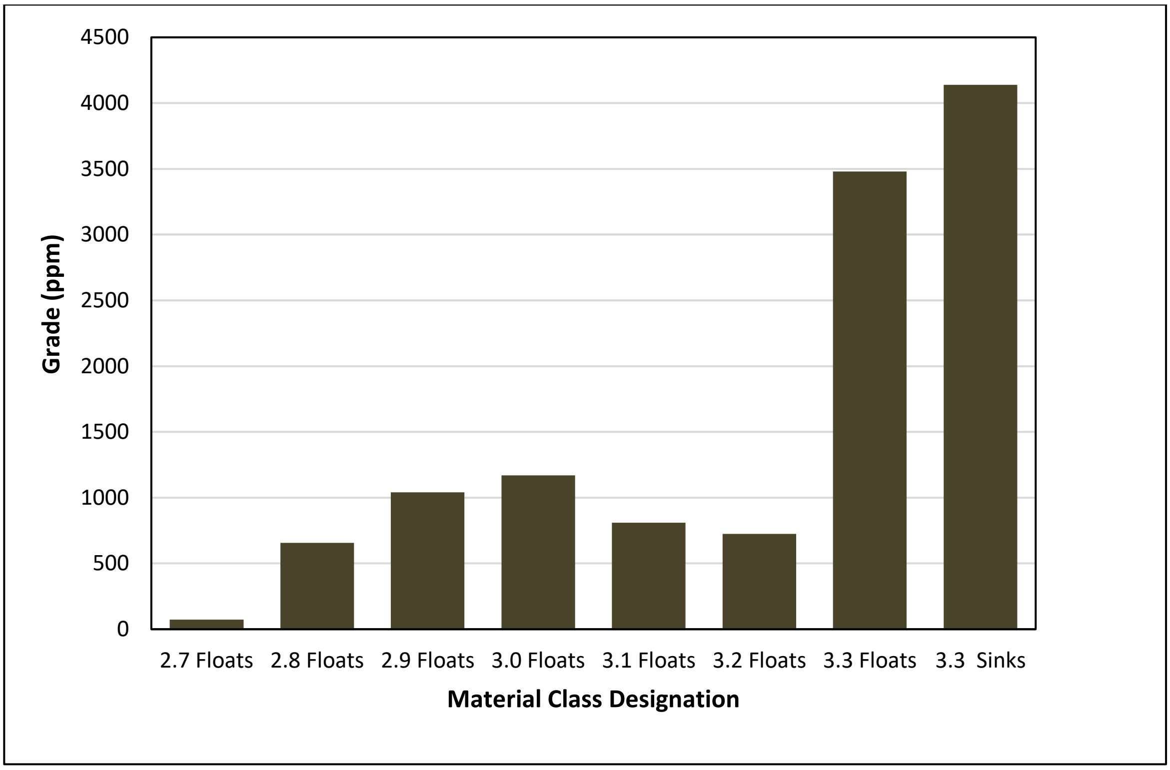


Figure 13-17

Sn grades of the HLS float samples



Cumulative mass recovery, as a function of separating SG, is presented in 13-18, while cumulative Li_2O grade and recovery are shown in Figures 13-19 and 13-20 respectively. Of immediate interest is the sudden decrease in cumulative Li_2O grade beyond a separating SG of 2.9. The cumulative grade at an SG of 2.9 is reported as 6.11 % Li_2O, which decreases to 4.46 % Li_2O at an SG of 2.8. This decrease is accompanied with an increase in Li_2O recovery of 6.96 %, in accordance with Figure 13-20.

It is clear from the data presented that the density **cut point should be specified at an SG of 2.9**, resulting in the production of a **concentrate containing 6.11 % Li_2O**, at a **Li_2O recovery of 74.85 %** and an **overall product yield or mass recovery of 10.22 %**.

With reference to previous mineralogical studies completed on the material, the lithium species in the low SG floats (and therefore the DMS waste) is largely in the form of poorly liberated spodumene grains, in addition to less dense lithium species such as petalite and cookeite. The iron-containing lithium species zinnwaldite is also likely to be present.

Figure 13-18

Cumulative mass recovery of the +425 µm HLS tests conducted on the Kamativi composite sample

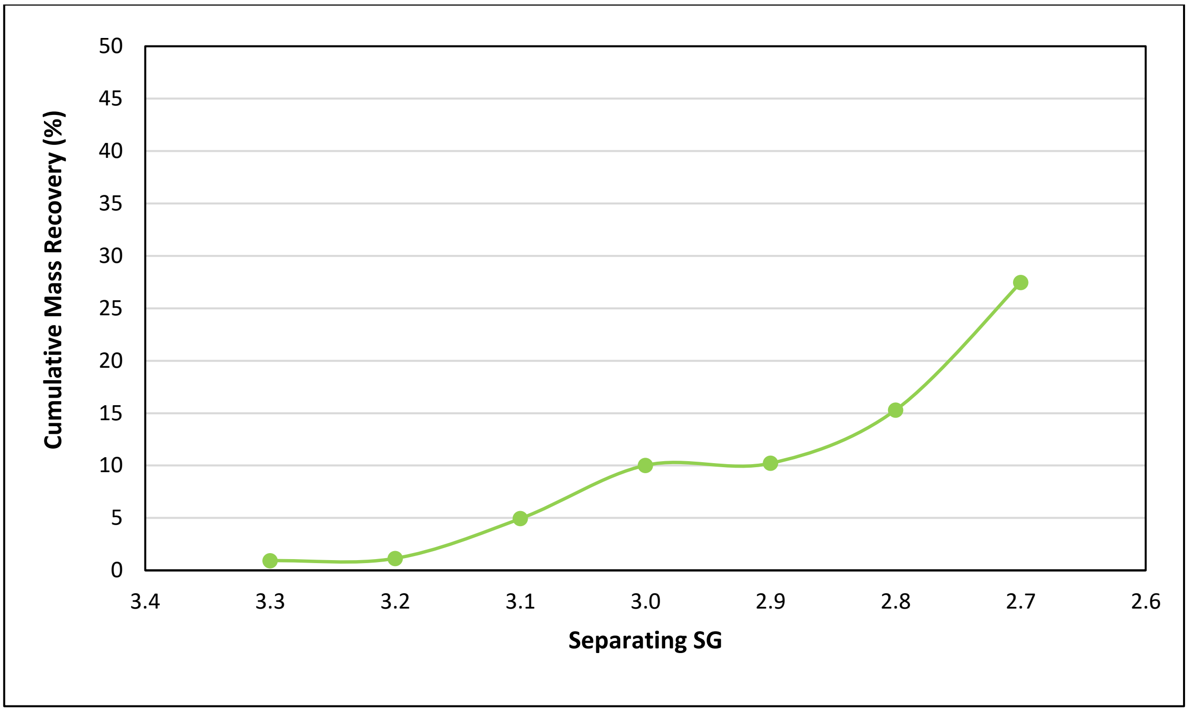


Figure 13-19

Cumulative Li_2O grades of the +425 µm HLS tests conducted on the Kamativi composite sample

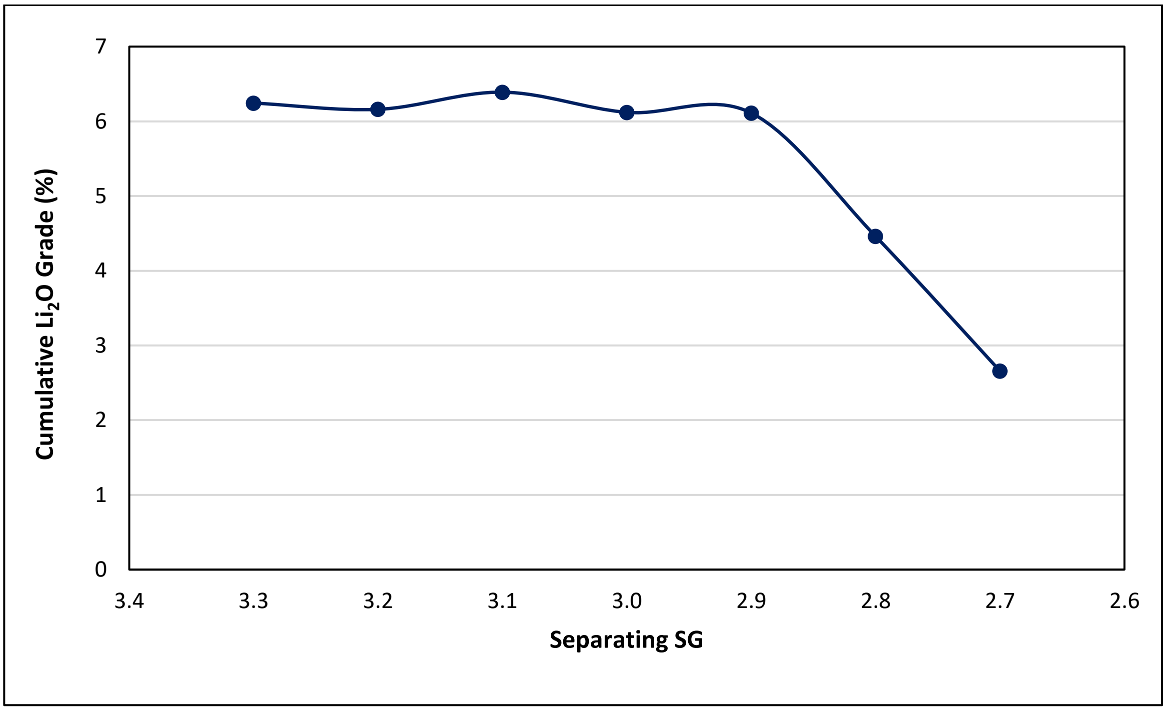
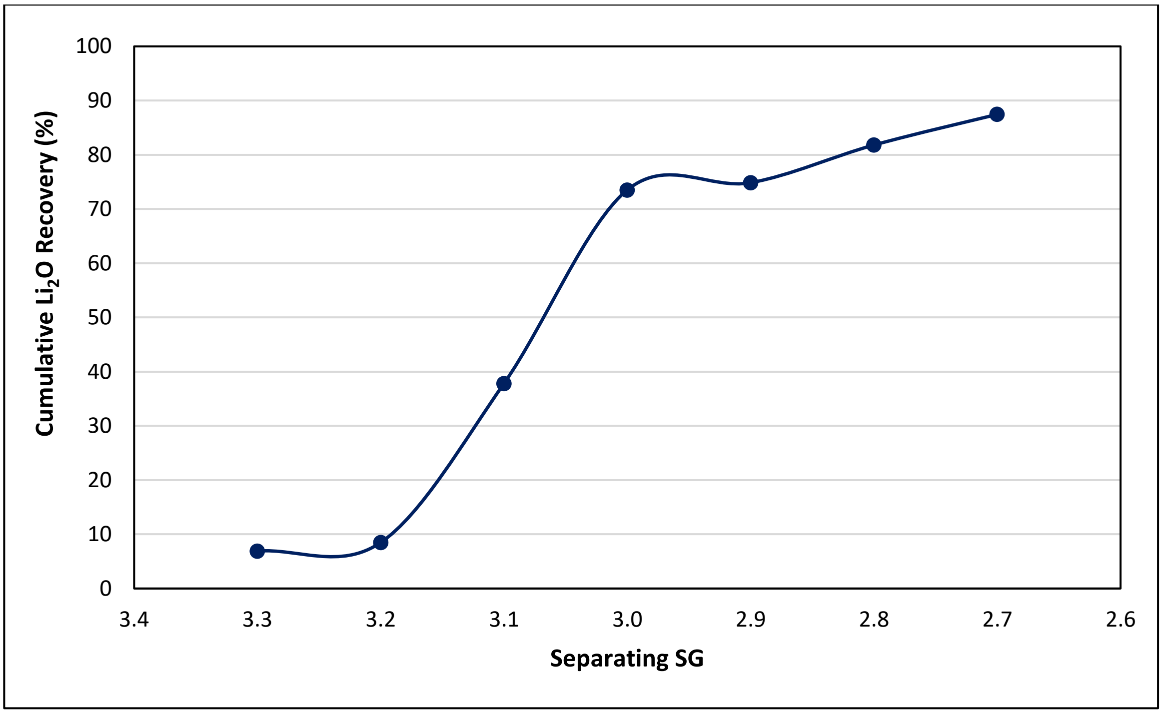


Figure 13-20

Cumulative Li_2O recovery of the +425 µm HLS tests conducted on the Kamativi composite sample



As discussed, the tin and iron grades and recoveries should also be considered. Cumulative tin grade and recovery curves are presented in Figures 13-21 and 13-22 respectively, with the equivalent data for iron being provided in Figures 13-23 and 13-24 respectively.

The proposed density cut point of 2.9 is exclusively based on the Li_2O grade data presented in 13-19. Therefore, with the cut point having been specified, the tin and iron grades and recoveries are fixed. **At an SG of 2.9, the cumulative tin grade is reported as 1144 ppm**, or approximately 0.11 %, representing an almost four-fold upgrading of the feed material in tin content. The **cumulative recovery of the element is determined as 38.74 %**. The significant increase in tin grade at higher densities indicates that the application of gravity separation methods may yield a heavy mineral concentrate containing tin, however further test work must be completed to verify this assertion. The low recovery is due to poor liberation of cassiterite grains, which report to the low density floats fractions.

The **cumulative iron grade and recovery, at the specified cut point are 0.93 % and 20 % respectively**. As discussed, iron is specified as a penalty element, with maximum iron grades relating to saleable spodumene concentrate being in the range of 0.1 to 0.2 % Fe. It is clear therefore, from the data presented, that the iron bearing minerals must be removed from the DMS concentrate material. This may be achieved through the use of magnetic separation equipment such as WHIMS, possibly used in conjunction with gravity separation technology such as tabling.

In accordance with a detailed mineralogical analysis preciously completed on the material, the iron species in the concentrate material are largely in the form of various iron-manganese oxides, zinnwaldite, garnet, tourmaline and iron phosphates.

Figure 13-21

Cumulative Sn grade of the +425 µm HLS tests conducted on the Kamativi composite sample

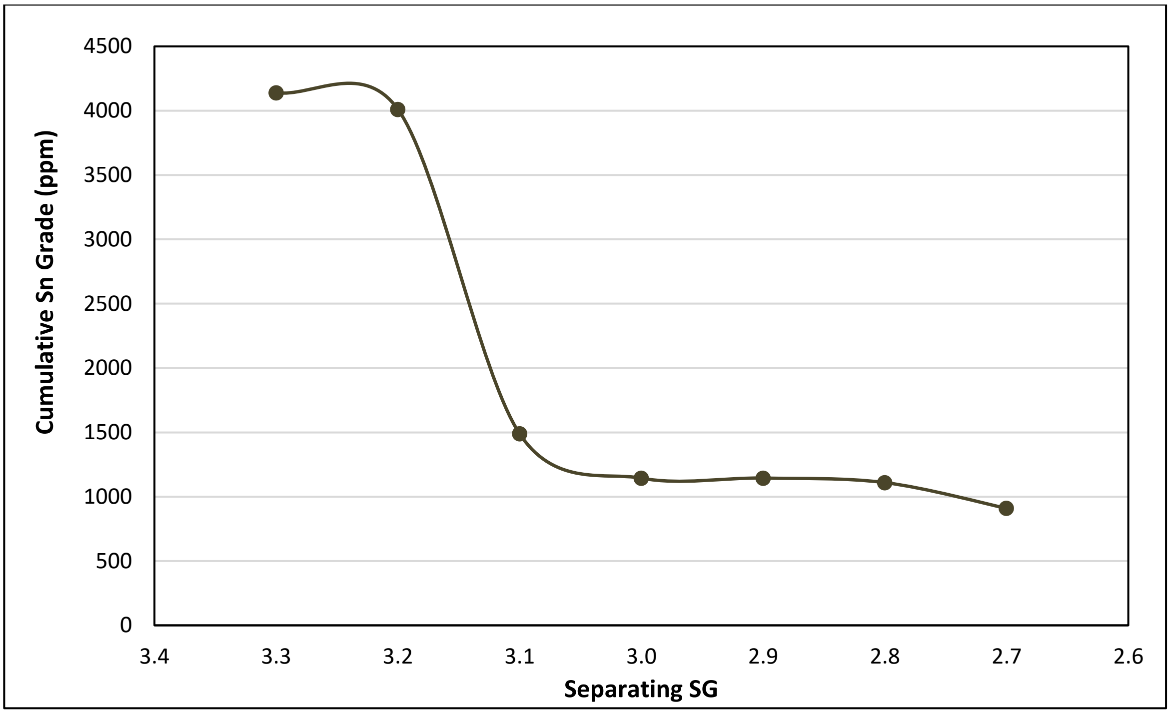


Figure 13-22

Cumulative Sn recovery of the +425 µm HLS tests conducted on the Kamativi composite sample

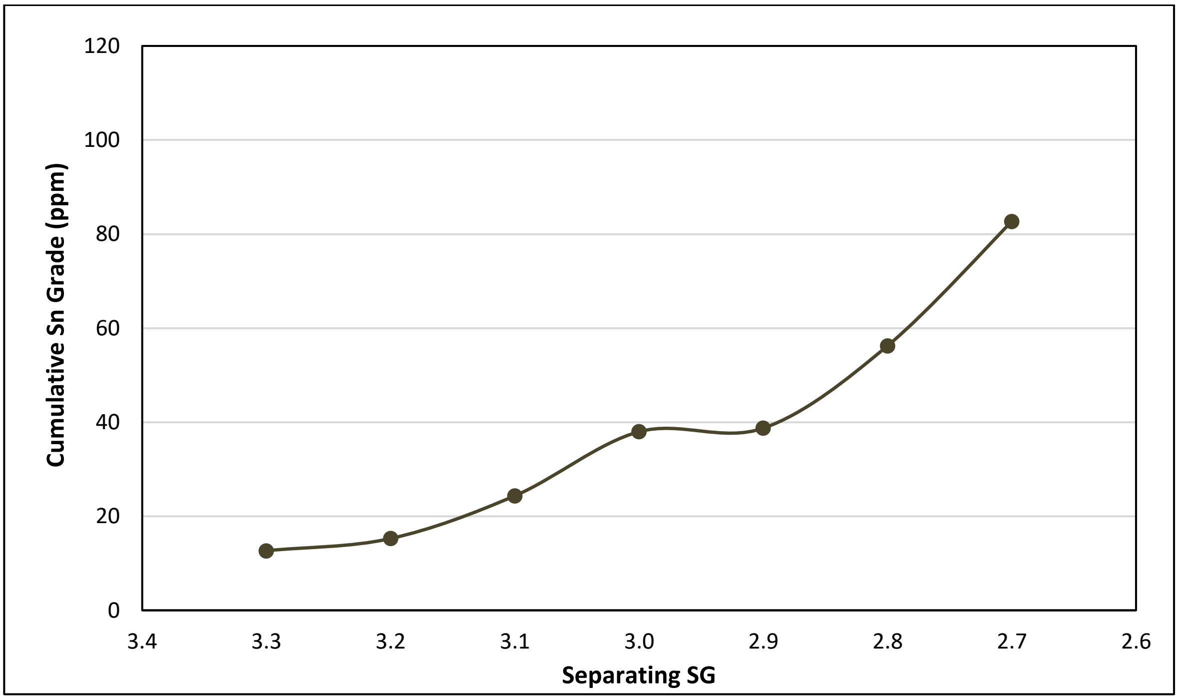


Figure 13-23

Cumulative Fe grade of the +425 µm HLS tests conducted on the Kamativi composite sample

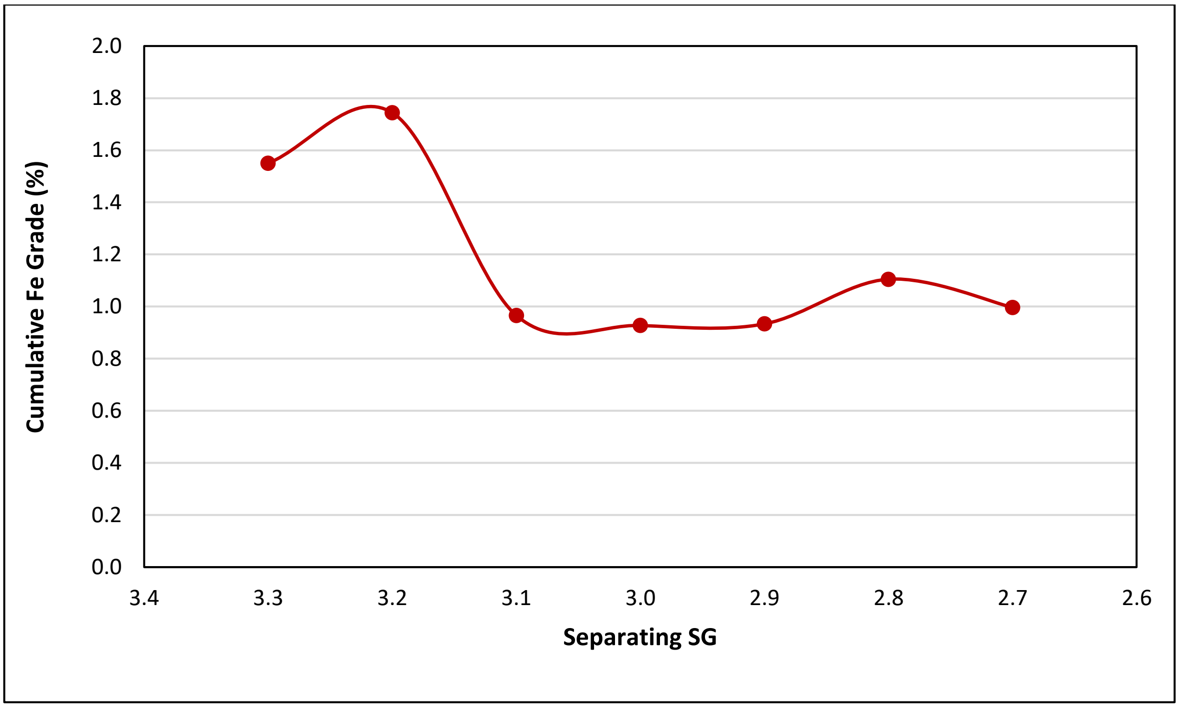
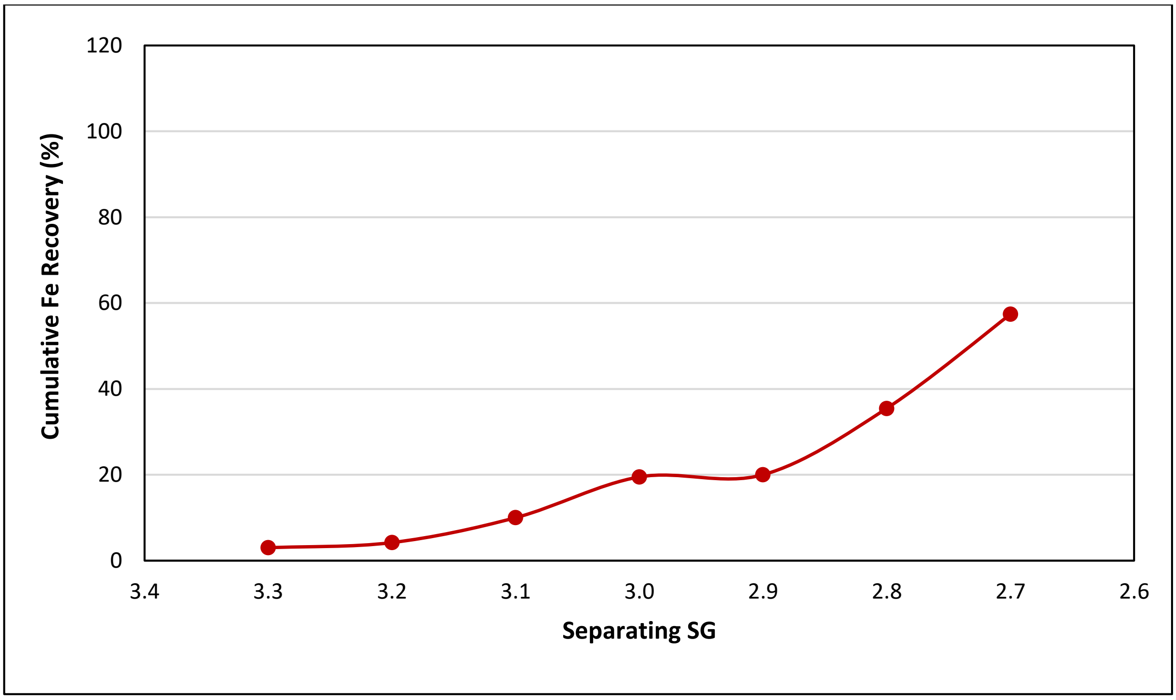


Figure 13-24

Cumulative Fe recovery of the +425 µm HLS tests conducted on the Kamativi composite sample



13.6 Scoping Magnetic Separation Test Work

This section of the document discusses the observations relating to the magnetic separation test work completed at Multotec's facilities in Spartan, Johannesburg.

Following the completion of the HLS test work, a concentrate sample was constituted from the various SG fraction samples, starting at the floats sample at SG 3 and ending with the sinks sample at SG 3.3.

As discussed, the sample feed grades were estimated as 6.11 % Li_2O and 0.93 % Fe. The concentrate sample was subsequently subjected to scoping magnetic separation test work, this for the purpose of reducing the iron content in the concentrate.

The feed material was initially separated using a rotary sample splitter, where one of the ten subsamples was set aside as a feed sample for analysis. The remaining nine subsamples were combined as feed to the magnetic separator.

The test work was conducted using a dry permanent roll magnetic separator, shown in Figure 13-25.

A number of different rougher tests were performed, using various magnetic rolls characterised by varying magnetic intensities. The mass yield to the magnetic fraction was measured at the end of each test run.

Figure 13-25

Dry permanent roll magnetic separator

The most effective test, conducted using a 100 mm magnetic roll at a belt thickness of 1 mm, achieved a mass pull to the magnetic fraction in the order of 5 %. The magnetic intensity was determined to be approximately 4700 gauss.

Photographs of the feed material and the rougher stage magnetic, non-magnetic and middlings fractions are shown in Figure 13-26.

The non-magnetic material was subsequently split into two subsamples using the sample splitter, where one sample was to be retained by CMPSA for analysis, while the other was charged to the magnetic separator as feed in a cleaning stage. The yield to the magnetics fraction was minimal and estimated to be in the order of 0.5 %. A photograph of the cleaner stage non-magnetic material is shown in Figure 13-27. This material was sub-sampled in addition to the rougher stage feed, magnetic and non-magnetic material fractions. A subsample of each material type was submitted to SGS for Li and Fe analysis by ICP and XRF.

Photographs of the rougher magnetic and cleaner non-magnetic fractions, viewed under a loupe, are shown in Figure 13-28. The remaining impurities in the non-magnetic fraction are likely non-magnetic mineral species, which may be identified through a detailed mineralogical study. The loss of spodumene crystals to the magnetic fraction is due primarily to static effects experienced by the particles during the test. The majority of the material in the magnetic fraction is likely to be in the form of tourmaline, garnet, zinnwaldite and iron manganese oxides.

Figure 13-26

Rougher magnetic separation test: feed (top centre), magnetic (bottom left), middlings (bottom centre) and non-magnetic (bottom right) material fractions

Figure 13-27

Cleaner test: non-magnetic fraction

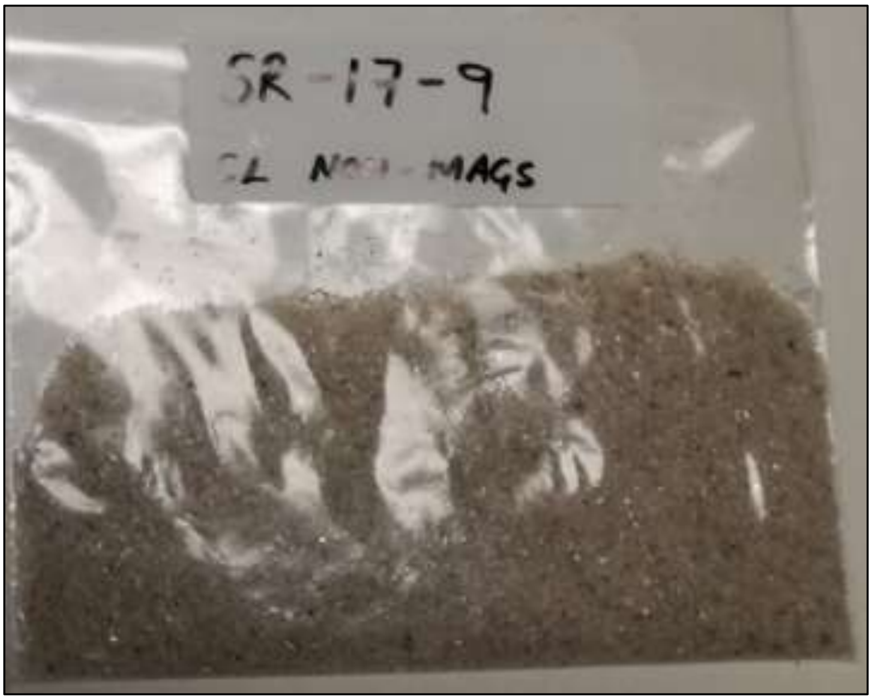


Figure 13-28

Cleaner test: non-magnetic fraction (left) and rougher test – magnetic fraction (right)

Li_2O and iron grades of the various material fractions associated with the rougher and cleaner tests conducted at approximately 4700 gauss, as discussed, are shown in Table 13-9. The assays were performed by SGS using ICP technology.

Table 13-9

Magnetic separation material fraction Li_2O and Fe grades

Material Fraction	Li_2O Grade (%)	Fe Grade (%)
Rougher Feed	6.848	0.83
Rougher Non-Magnetics	7.127	0.45
Rougher Magnetics	1.376	9.96
Cleaner Non-Magnetics	7.084	0.26
Rougher Feed (Calculated)	6.839	0.926

The data presented in Table 13-9 illustrates that removal of the iron bearing species may be achieved, as evidenced by the decrease in calculated iron grade from 0.926 % in the feed material, to 0.26 % in the cleaner non-magnetic fraction. The decrease in product mass, coupled with minimal loss of lithium bearing material to the magnetic fraction, results in an increase in Li_2O grade observed in the non-magnetic fractions.

The estimated mass yields and recoveries of Li_2O and iron to each fraction associated with the rougher test are shown in Table 13-10. Approximately 54 % of the iron is recovered in the rougher stage magnetic fraction with a corresponding Li_2O loss of only 1 %.

In accordance with the data presented in Table 13-11, a stage recovery of 42.51 % of the remaining iron is recovered in the cleaner stage magnetic fraction, producing a final concentrate which exhibits grades of 7.08 % Li_2O and 0.26 % Fe (or 0.37 % Fe_2O_3), in accordance with Table 13-9. Overall iron recovery can be calculated as 73.44 % at a Li_2O loss of 2.09 %, which is due predominantly to the presence of zinnwaldite in the magnetic fraction and the loss of spodumene to the same material due to static effects.

Table 13-10

Magnetic separation material fraction Li_2O and Fe grades, rougher test

Material Fraction	Mass Yield (%)	Li_2O Recovery (%)	Fe Recoveryv(%)
Rougher Feed	100.00	100.00	100.00
Rougher Non-Magnetics	95.00	99.00	46.19
Rougher Magnetics	5.00	1.00	53.81

Table 13-11

Magnetic separation material fraction Li_2O and Fe grades, cleaner test

Material Fraction	Mass Yield (%)	Li_2O Recovery (%)	Fe Recoveryv(%)
Cleaner Feed	100.00	100.00	100.00
Cleaner Non-Magnetics	99.50	98.90	57.49
Cleaner Magnetics (Calculated)	0.50	1.10	42.51

13.7 HLS and Magnetic Separation Test Work Summary

Heavy liquid separation test work completed by SGS has concluded the following:

- 46.4 % of the feed material is contained within the coarser +425 µm size fraction, however due to a higher Li_2O grade associated with this coarser fraction (in the range of 0.85 % Li_2O), in the order of 55 % of the lithium is contained within this fraction.
- The production of a high grade lithium concentrate may be optimally achieved at a relative cut density or SG of 2.9.
- The concentrate produced at this SG, under ideal conditions is characterised by the following ***calculated*** parameters:
    - Li_2O grade of 6.11 % at a recovery of 74.85 %
    - Sn grade of 1144 ppm at a recovery of 38.74 %
    - Fe grade of 0.93 % at a recovery of 20 %
    - A total mass yield or recovery of 10.22 %
- The removal of iron species is essential due to concentrate iron grades which exceed those specified for a saleable concentrate.

Based on data associated with the initial magnetic separation scoping test, the following is concluded:

- 73.44 % of the iron was removed from the heavy liquid separation concentrate, with a corresponding Li_2O loss of 2.09 %.
- The final concentrate assay was recorded as 7.08 % Li_2O and 0.26 % Fe.

13.8 Future Work

Further test work will be conducted after collecting a large, compliant, and representative sample from the Kamativi tailings resource. This test work, with the relevant data being used as inputs for flowsheet design, will be carried out according to the following individual test campaigns:

- dense media separation pilot test work;
- magnetic separation test work; and
- flotation test work.

Dense media separation pilot scale test work will be conducted on a coarser fraction of the tailings material, that will be taken in a compliant manner, to ascertain the feasibility of the production of a spodumene pre-concentrate through the application of dense media separation technology. Magnetic separation test work will likely be required to remove iron-bearing species from the HLS spodumene concentrate. Test work will involve the processing of concentrate material at various magnetic intensities, for the purposes of determining optimum operating parameters for the magnetic separation unit.

Due to the low efficiencies realised during the processing of fine material in a DMS circuit, flotation test work is to be completed on the finer fraction of the tailings material. Optimum operating conditions and reagent addition will be determined, in addition to the number of processing stages required. Grind optimisation and lock cycle flotation will also be conducted.

A bulk sample is required for final pilot phase test work and optimisation, which is to include the processing of the material using a flotation characterisation test rig ("FCTR"). It is currently proposed that this sample will be assembled from six trenches to be excavated across the dam surface to a depth of 2 m. Approximately 180 t of bulk material will be collected, blended and sub-sampled to provide the 500 kg requirement. The remaining material will be retained for possible future test work.

13.9 Tailings Disposal

Environmental Consultants MsMontan and Green Resource Company, visited the site as part of the environmental fatal flaw review. During the site visit three potential site options were considered:

- the first two site options consider the opportunity to minimise the footprint of any new tailings facility by concurrently scheduling mining of existing tailings and deposition of the new tailings onto mined-out areas of the existing dump footprint; and
- the third option is an area situated in a valley to the north of the existing dump.

The third option is currently the recommended site as there appears to be nothing that would hinder the establishment of the facility.

Further detailed site selection and design in conjunction with an Environmental Impact Assessment will be conducted during the feasibility study to determine final site selection.

14 MINERAL RESOURCE ESTIMATES

On behalf of ZIM, MSA has completed a maiden Mineral Resource estimate for the Kamativi Lithium Tailings Project ("Kamativi").

To the best of the Qualified Person's knowledge there are currently no title, legal, taxation, marketing, permitting, socio-economic or other relevant issues that may materially affect the Mineral Resource described in this Technical Report.

The Mineral Resource estimate incorporates drilling data collected by ZIM from March 2018 to June 2018, which, in the Qualified Person's opinion, were collected in accordance with The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Exploration Best Practices Guidelines".

The Mineral Resource was estimated using the 2003 CIM "Best Practice Guidelines for Estimation of Mineral Resources and Mineral Reserves" and classified in accordance with the "2014 CIM Definition Standards". It should be noted that Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Mineral Resource estimate was conducted using Datamine Studio 3 software, as well as Sequent Leapfrog Geo for wireframe construction, and Microsoft Excel and Snowden Supervisor for data analysis. The Mineral Resource estimate was completed by Mrs Ipelo Gasela (Pr. Sci. Nat.) who is the Qualified Person for this Mineral Resource.

14.1 Database

The geochemical data that this estimate is based on was collected through core and auger drilling of the Kamativi tailings dump from March 2018 to June 2018. The cut-off date for the data used for this estimate is the 20th August 2018. As at the cut-off-date, there were no outstanding data of relevance to this estimate and the database was complete.

A total of 116 holes were drilled by independent drilling contractors. The drill holes comprise of 61 core holes and 55 auger holes. One hole was not assayed because it went into rock beneath tailings in the first run. Therefore, 115 drill holes were used to estimate the Mineral Resource. The drill holes were completed on a 100 m grid, with some areas to the south west being drilled at closer spacing. The drilling programme started with core drilling and changed to auger drilling due to slow drilling rates and low recoveries experienced from the core drilling method. Most of the core drilling was carried out towards the south of the tailings dump, and auger drilling was mostly undertaken in the northern area of the dump.

The data from the drilling programme and pitting (for density measurements) are stored in a Microsoft Access database. The database is comprehensive and includes collar surveys, lithology, density and quality control data. Where the assays returned a value of less than the lower detection limit, half the lower detection limit was assigned for estimation purposes.

In the Qualified Person's opinion, the Microsoft Access database conforms to modern acceptable database management protocols and is sufficient for storing data for Mineral Resource estimation purposes.

14.2 Exploratory Analysis of Raw Data

14.2.1 Validation of the data

A validation process, which included the following checks, was undertaken for Mineral Resource estimation purposes:

- Examining the sample assay, collar, and geology data to ensure that the data are complete for all drill holes.
- Examining the desurveyed data in three dimensions to check for gross spatial errors.
- Examination of the assay and density data in order to ascertain whether they are within expected ranges.
- Checks for "FROM-TO" errors, to ensure that the sample data do not overlap one another or that there are no unexplained gaps in the data.

The data validation process revealed the following:

- All drill holes were assayed except for one, namely KT90. No assay data occur where no sample material was recovered.
- All intercepts were logged as either "tails" or "bottom of dump".
- Minor overlaps were observed towards the end of some of the holes. These were resolved by calculating weighted lengths based on mass recovered.
- No spatial errors were observed when examining the desurveyed data in three dimensions.
- The density values range between 1.09 t/m^3 and 2.08 t/m^3. These density values are considered to be within acceptable limits for tailings material of this nature and no density data were rejected for the purpose of Mineral Resource estimation.

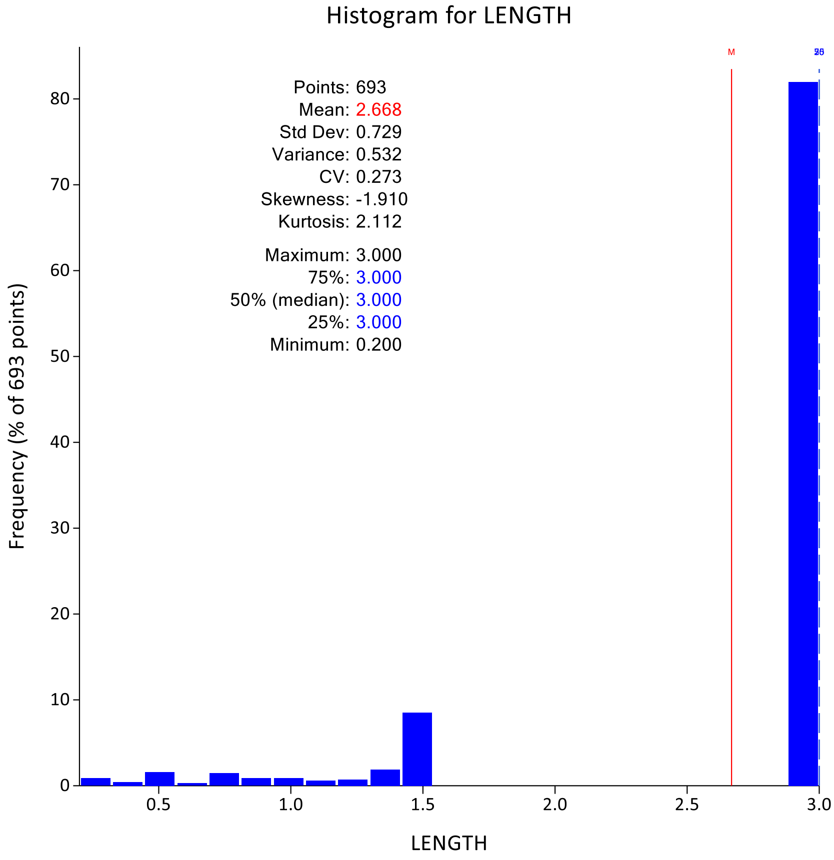
The data passed the validation checks and the Qualified Person considers the data to be acceptable for use in Mineral Resource estimation.

14.2.2 Statistics of Sample length

Samples collected from the tailings material range from 0.3 m to 1.5 m in length, with the most common sample length being 1.5 m, which is the length of the drill rods used. Shorter samples were collected towards the base of the dump.

Consecutive samples collected from the drilling were composited into 3 m samples for assaying. Successive samples that could not be composited to 3 m samples were assayed without being composited, hence there are assayed samples shorter than 3 m (Figure 14 1).

Figure 14-1
Histogram and statistics of Kamativi composite sample length



Source: MSA 2018

14.2.3 Statistics of Assay Data

Samples were assayed for Li, Sn, Ta, Fe and Nb, amongst other elements, and were converted to oxides using the factors shown in Table 14-1. Summary statistics of the sample assays for Li_2O, SnO_2, Ta_2O_5, Fe_2O_3 and Nb_2O_5 are presented in Table 14-2.

Table 14-1
Factors used to convert elements to oxides

Element	Conversion factor	Oxide
Li	2.1530	Li_2O
Sn	1.2696	SnO_2
Ta	1.2211	Ta_2O_5
Fe	1.4297	Fe_2O_3
Nb	1.4305	Nb_2O_5

The sample grades for Li_2O range from 0.12% to 1.11%, and the mean grade is 0.58%. All the variables analysed show low variability with coefficient of variation values of between 0.29 and 0.53 (Table **14-2**).

Table 14-2
Sample assay statistics

Variable	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation
Li_2O (ppm)	1,195	11,088	5,843	2,063	0.35
SnO_2 (ppm)	14	1,371	500	153	0.30
Ta_2O_5 (ppm)	8	278	39	21	0.53
Fe_2O_3 (%)	0.5	9.0	1.3	0.6	0.44
Nb_2O_5 (ppm)	23	1,774	64	19	0.29

14.2.4 Bivariate Statistics

Bivariate statistics were calculated on sample assay grades to assess if there is any correlation between the variables that requires consideration during grade estimation. The correlation matrix in Table 14-3 shows the linear correlation coefficient for all the variable pairs. These demonstrate that the attributes are poorly correlated with one another.

Table 14-3
Correlation matrix of sample assays

	Li_2O	SnO_2	Ta_2O_5	Fe_2O_3	Nb_2O_5
Li_2O	-	0.09	-0.24	-0.10	-0.53
SnO_2	0.09	-	0.10	0.41	0.20
Ta_2O5	-0.24	0.10	-	0.01	0.56
Fe_2O_3	-0.10	0.41	0.01	-	0.04
Nb_2O_5	-0.53	0.20	0.56	0.04	-

14.2.5 Density statistics

A total of 214 density measurements were taken from 60 pits excavated at different locations over the dump. The density values range from 1.09 t/m^3 to 2.08 t/m^3 and the average of the density values is 1.67 t/m^3 (Table 14-4).

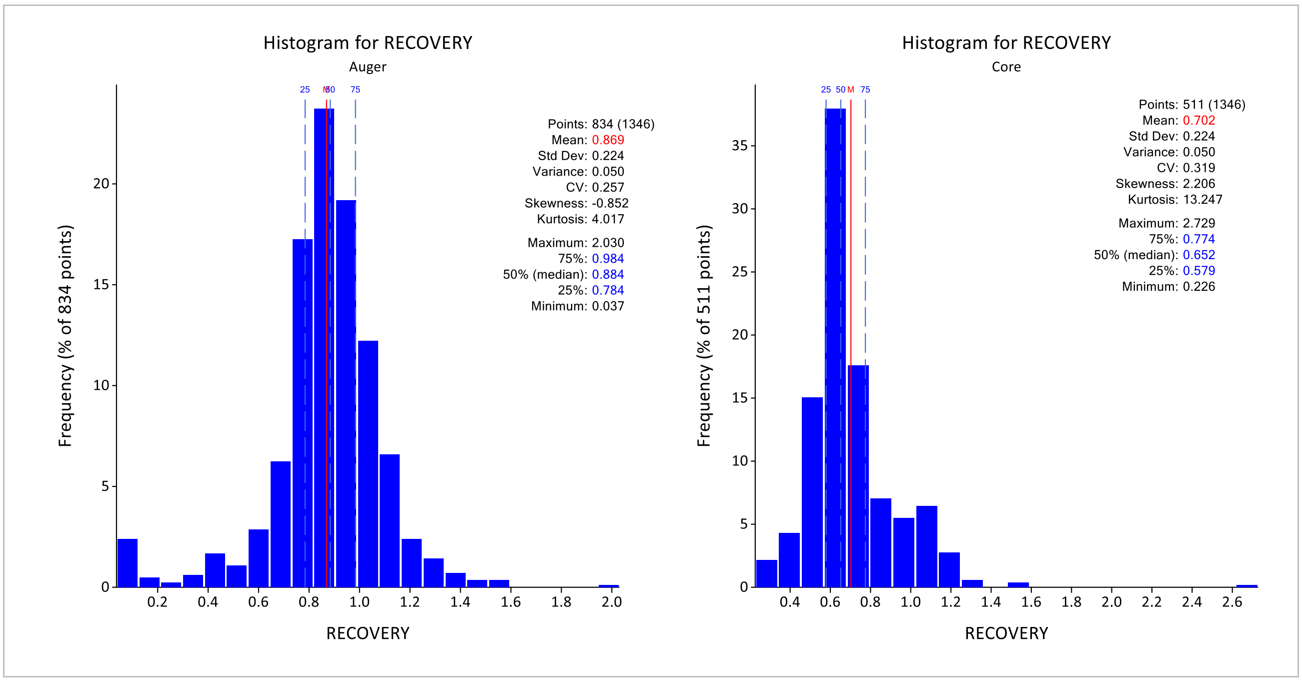
Table 14-4
Density statistics

Variable	Minimum	Maximum	Mean	Standard deviation	Coefficient of variation
Density	1.09	2.08	1.67	0.18	0.11

14.2.6 Recoveries

Recovery was assessed for the different drilling methods in order to gauge the representivity of the samples. The auger method gave acceptable average recovery (87%), while the coring method gave a lower average recovery (70%). Several samples have recovery of greater than 100%, which is considered to be due to hole collapse (Figure 14-2).

Figure 14-2
Recovery histograms by drillhole type at Kamativi



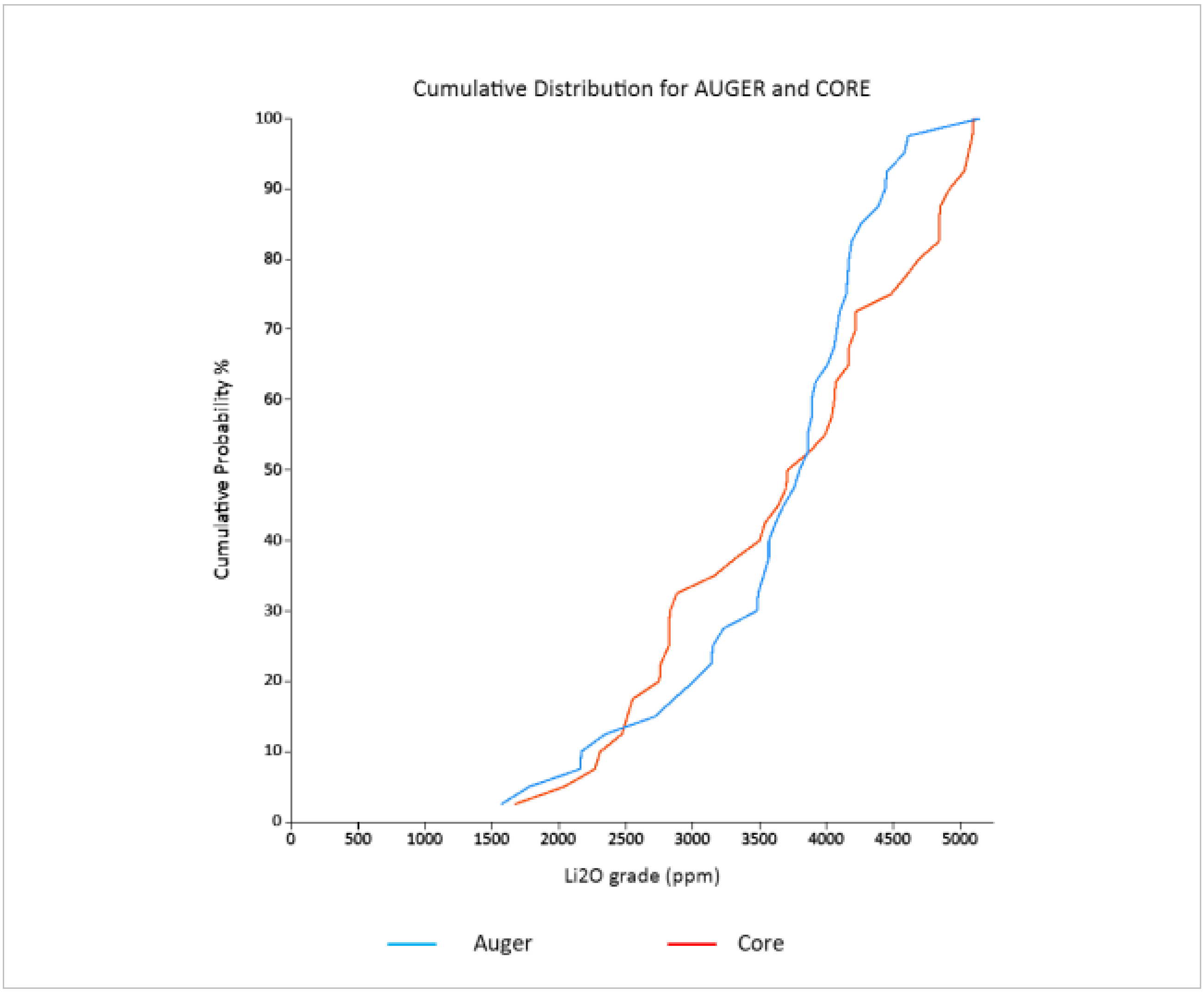
Source: MSA, 2018

14.2.7 Drilling methods bias test

Since different drilling methods were utilised at the Kamativi tailings dump, a bias analysis was undertaken to determine if the sample assays from the two drilling methods are consistent with one another. Seven core holes that did not penetrate to the base of the dump were twin drilled with auger drilling. The results from the corresponding intervals of the twinned holes were interrogated using cumulative distribution plots in order to determine the similarity of the data acquired through the different methods (Figure 14-3). The twinned intervals demonstrate similar grade distributions, although it was noted that the core drilling exhibits higher grade variability. In the QP’s opinion, the data collected using the two methods do not show significant bias with respect to one another and are appropriate to use together in Mineral Resource estimation.

Figure 14-3

Cumulative distribution plot of the twinned hole sample assay data for Li_2O grades



Source: MSA 2018

14.2.8 Summary of exploratory analysis of the raw dataset

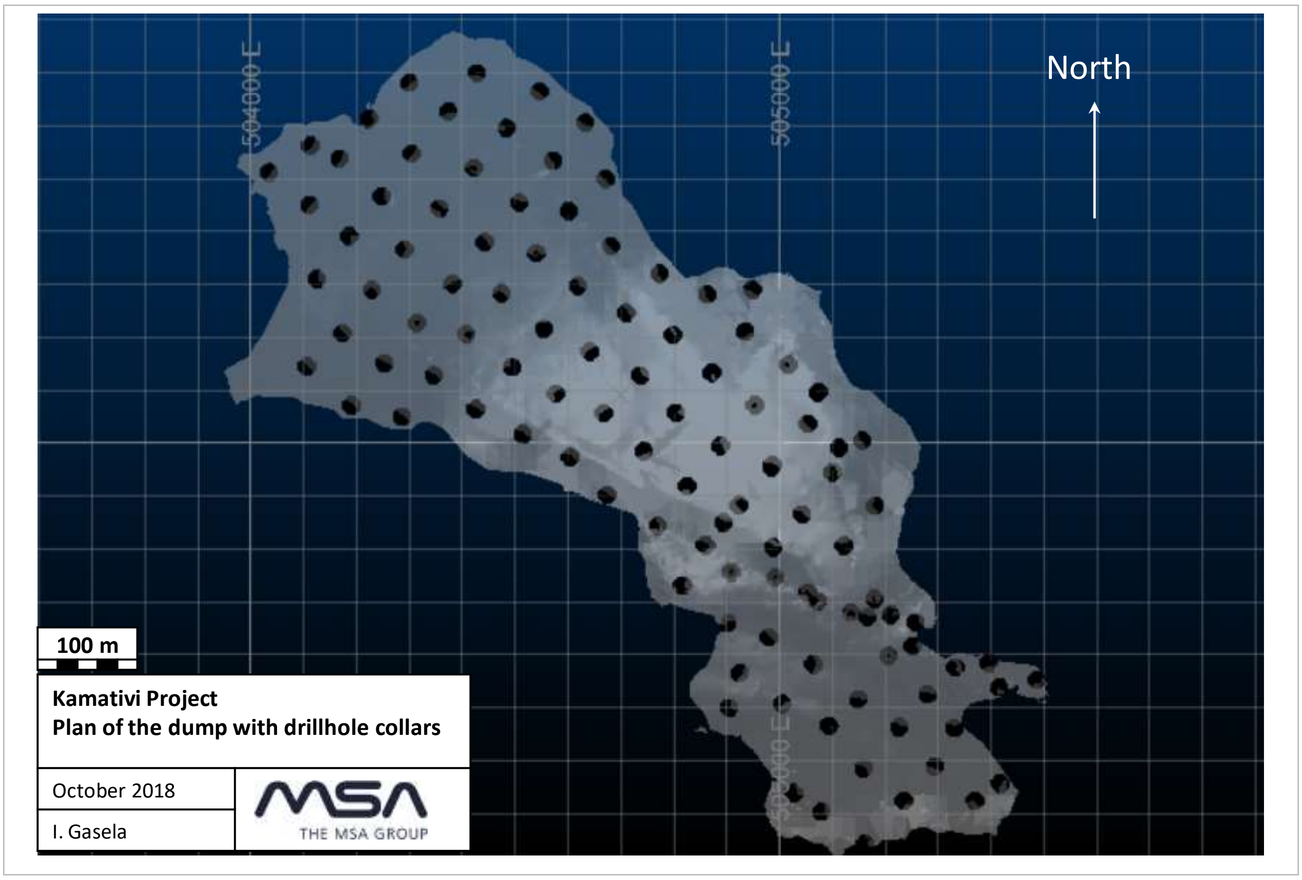
- The drilling information is suitably stored in a Microsoft Access database.
- The data passed the validation checks, following minor corrections.
- All recovered intersections were sampled (except for drillhole KT90) and have geochemical analyses.
- All sample assay and density values were accepted for Mineral Resource estimation.
- None of the assayed attributes are correlated with one another.
- There is no significant bias between the two drilling methods used to evaluate the dump.

14.3 Geological Modelling

A three-dimensional wireframe of the dump was created by merging two wireframe surfaces representing the top and bottom of the dump. The top of the dump surface is based on surveyed points on the surface of the dump and around the boundary of the dump, as well as drillhole collar surveys. The surface representing the bottom of the dump is based on drillhole intersections of the base, as well as survey points of the boundary of the dump. There are two instances where the modelling favours the surveyed boundary points over the drillhole intersection of the bottom of the dump. These are intersections close to the dump edge and no significant impact on the volume determination of the dump will arise. Not all of the drill holes penetrated the base of the dump. An additional surface was modelled representing the base of the drilling, where drill holes did not reach the base.

The Kamativi tailings dump has dimensions of approximately 1,900 m northwest to southeast and 580 m northeast to southwest, with a maximum height of 39 m (Figure 14-4 and Figure 14-5). A total dump volume of approximately 16.0 million cubic metres was estimated based on the model.

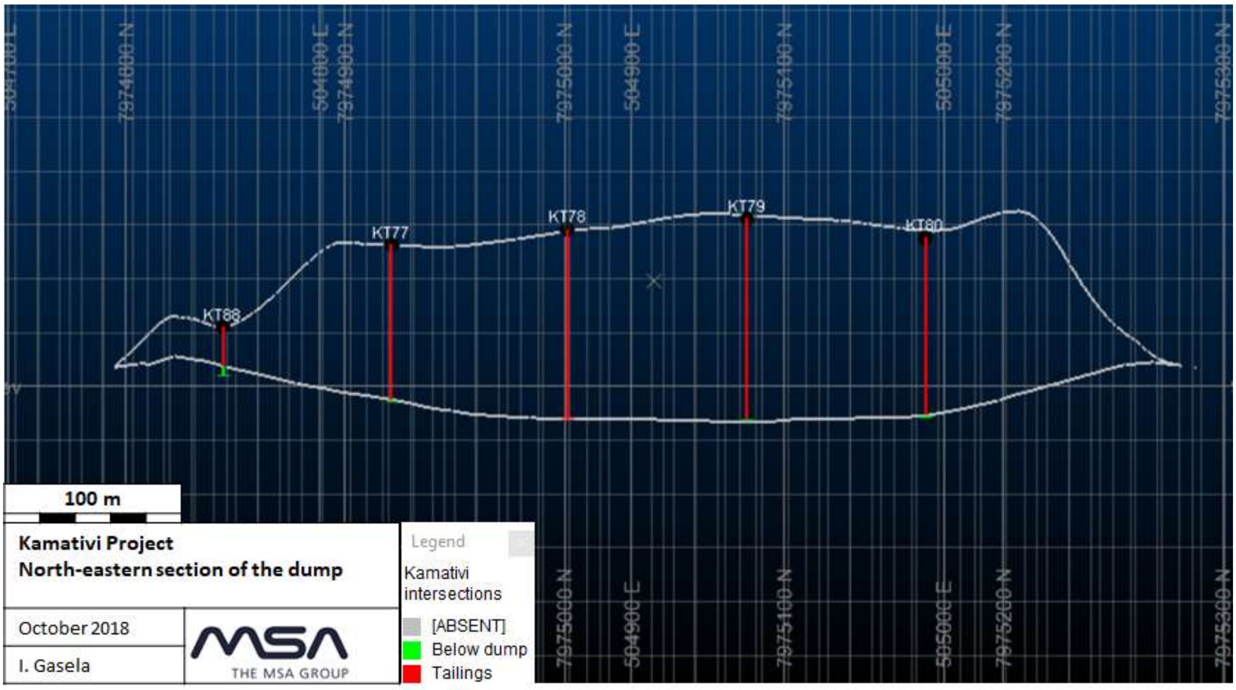
Figure 14-4
Plan of the dump with drillhole collars



Source: MSA 2018

Figure 14-5

Northeast to southwest cross section of the dump. Three times vertical exaggeration.



Source: MSA 2018

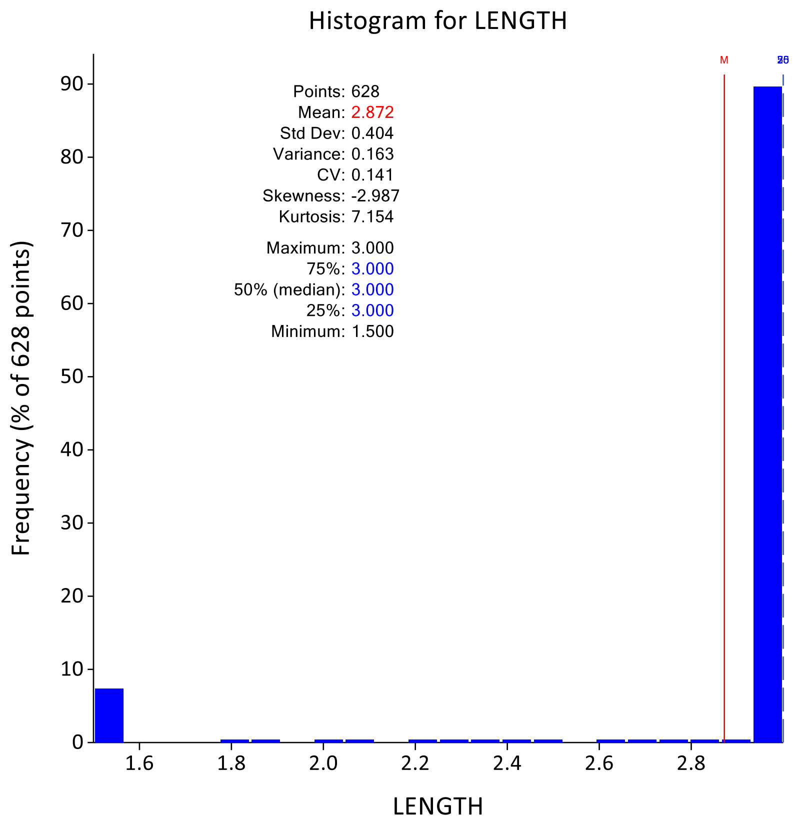
14.4 Estimation Domains

The tailings were estimated as a single domain.

14.5 Compositing

Since assaying was on 3 m composite samples, this length was used for sample composites for grade estimation. Some of the sample lengths were shorter than 3 m towards the end of the hole, and a minimum length of 1.5 m was applied, with any sample shorter than that being discarded. A total of 34 tailings samples shorter than 1.5 m were discarded. The discarded composites were similar in grade to the rest of the composites, thus discarding these short composites did not introduce bias. A histogram of the composited assayed sample lengths is shown in Figure 14-6.

Figure 14-6
Histogram and statistics of composited lengths



Source: MSA, 2018

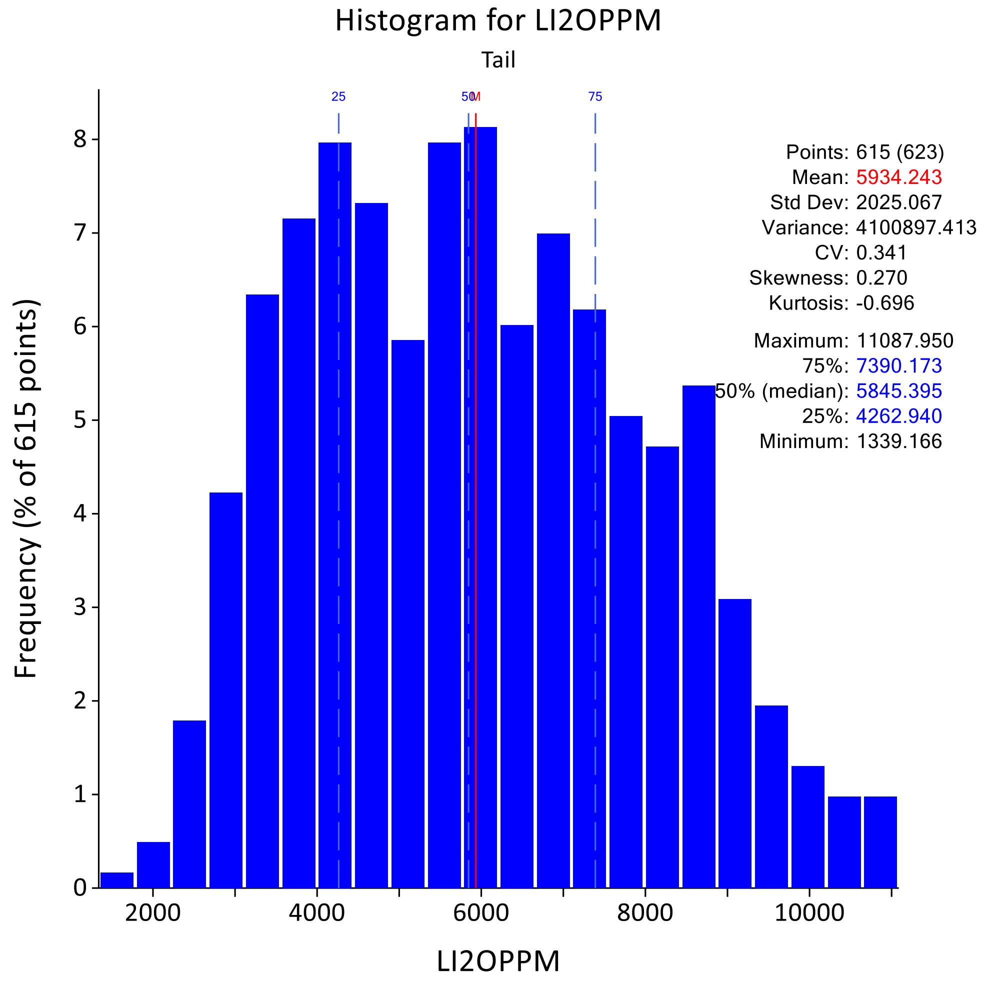
14.5.1 Composite statistics

Statistics of the composited tailings sample assays are shown as a histogram in Figure 14-7, and the summary statistics are presented in Table 14-5. The Li_2O grade distribution is slightly positively skewed and has a coefficient of variation (CV) of 0.34. The histogram indicates mixed distributions. The relatively low Li_2O grades occur in the northern- and southern-most part of the dump, while the middle of the dump has relatively high grades. The higher-grade population transitions into the low-grade areas and hence they were not treated as separate estimation domains.

SnO_2, Ta_2O_5, Fe_2O_3 and Nb_2O_5 distributions are positively skewed with CVs of approximately 0.30, 0.53, 0.27 and 0.29 respectively.

Figure 14-7

Histogram and statistics of composited Li_2O grades for tailings



Source: MSA 2018

Table 14-5

Tailings composite statistics for other variables

Variable	Minimum	Maximum	Mean	Standard deviation	Coefficient of variation
Li_2O (ppm)	1,339	11,088	5,934	2,025	0.34
SnO_2 (ppm)	14.0	1,371	498	148	0.30
Ta_2O_5 (ppm)	8.1	278	40	21	0.53
Fe_2O_3 (%)	0.5	4	1.2	0.3	0.27
Nb_2O_5 (ppm)	22.9	159	64	18	0.29

14.6 Geostatistical Analysis

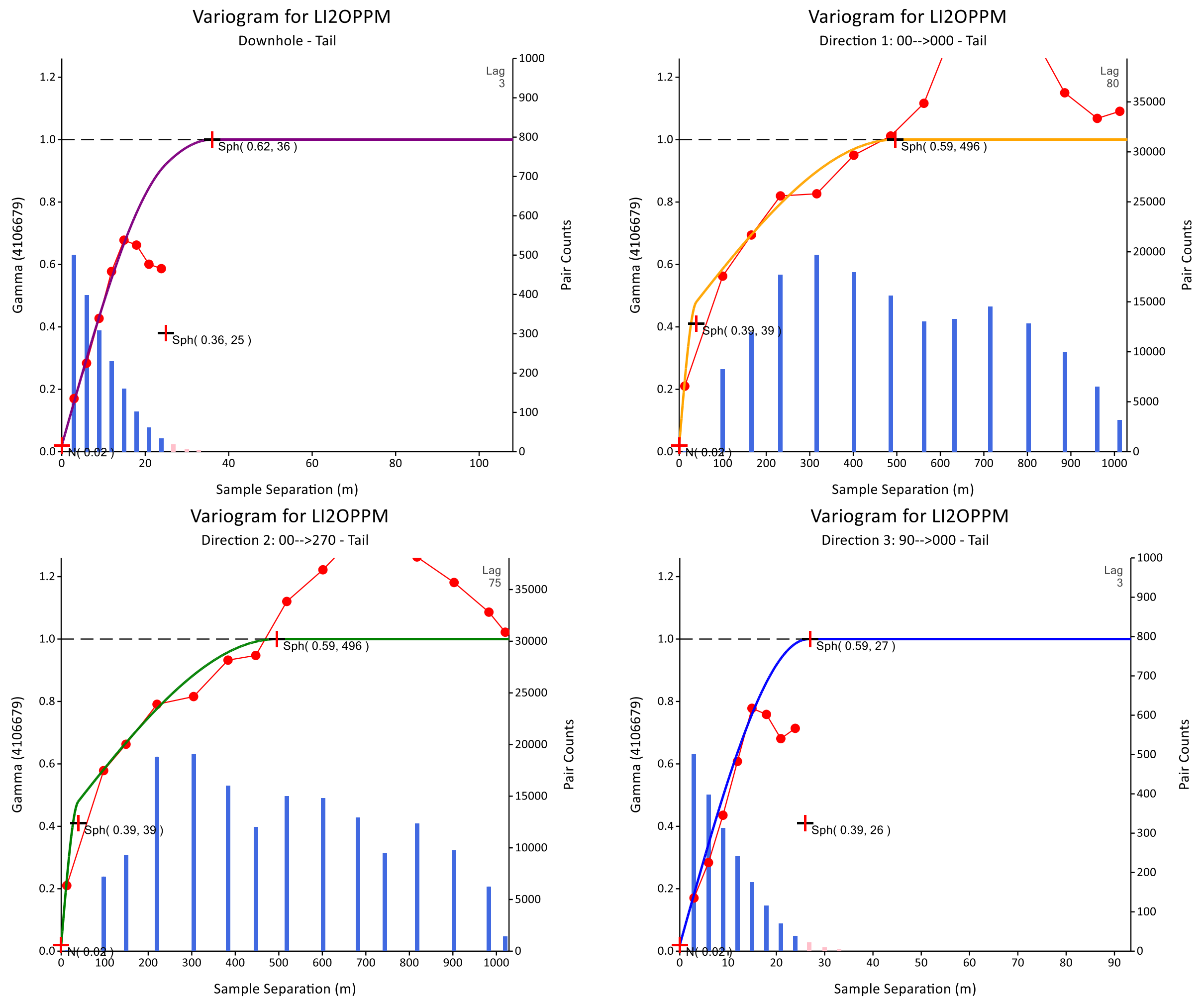
14.6.1 Variograms

Variograms were calculated on the three metre composites in the horizontal plane, since the tailings were deposited and settled horizontally. Variograms were modelled using two structures for Li_2O, SnO_2, Ta_2O_5 and Nb_2O_5 with isotropic ranges of between 200 m and 500 m in the horizontal plane and relatively short-ranges of between 20 m and 36 m vertically (Figure 14-8). The Fe_2O_3 variogram exhibited anisotropy in the horizontal plane and was modelled with the major direction at 030°/00° with a range of 327 m and the semi-major direction at 120°/00° with a range of 178 m (Table 14-6).

Table 14-6
Variogram parameters

Variable	Nugget effect	1st structure sill	1st structure ranges			2nd structure sill	2nd structure ranges		
			R1	R2	R3		R1	R2	R3
Li_2O	0.02	0.39	39	39	26	0.59	496	496	27
SnO_2	0.35	0.24	49	49	25	0.41	215	215	26
Ta_2O_5	0.33	0.09	273	273	19	0.58	377	377	21
Fe_2O_3	0.35	0.16	132	134	31	0.49	327	178	32
Nb_2O_5	0.22	0.21	326	326	24	0.57	513	513	32

Figure 14-8
Variogram for Li_2O grade for Kamativi tailings



Source: MSA 2018

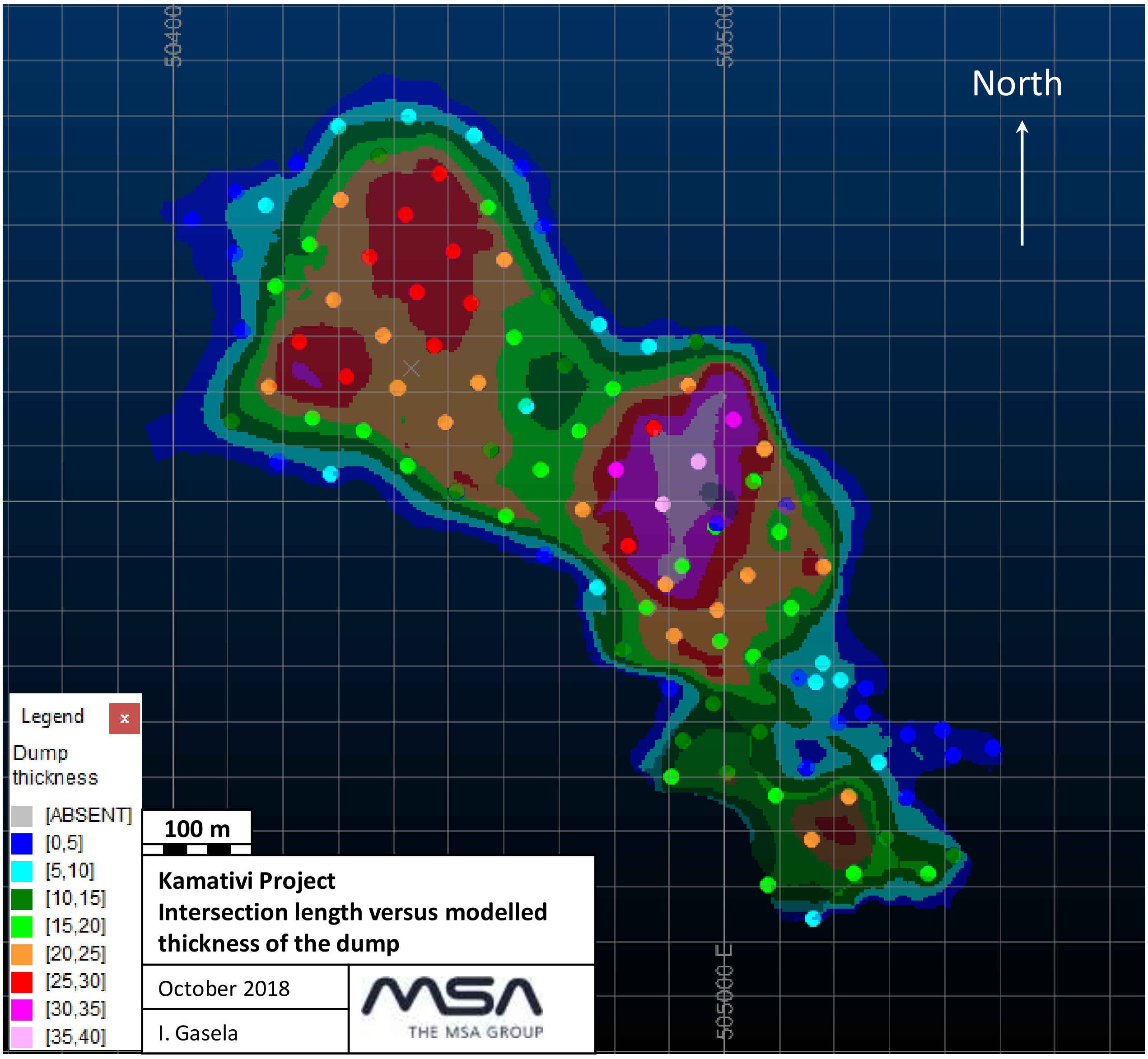
14.7 Block model

A block model was constructed within the tailings model with cell dimensions of 50 mX by 50 mY by 5 mZ. The block size is half the drillhole spacing, of approximately 100 m. The parent cells were sub-celled to 5 mX by 5 mY by 1 mZ in order to appropriately represent the volume of the wireframe.

In order to validate the model, the tailings wireframe was filled with single blocks and the vertical block thickness was compared to the drillhole length within the tailings. The modelled thicknesses are consistent with drillhole intersection lengths (Figure 14-9), demonstrating that the modelling of the tailings is appropriate.

Figure 14-9

Plan view of the drillhole intersection length and modelled dump thickness



Source: MSA 2018

14.8 Estimation

Li_2O, SnO_2, Ta_2O_5, Fe_2O_3 and Nb_2O_5 grades were estimated into the model parent cells using ordinary kriging. A minimum number of four composites was required to estimate a block, and the maximum number allowed was twelve. A maximum of three composites were allowed from the same hole.

The same search parameters were used to estimate all the grade variables, as shown in Table 14-7. A three-pass search strategy was used for estimation, with the first search range at 100 mX by 100 mY by 5 mZ, which was chosen as the holes were drilled on a regular 100 m grid. The second search was 1.5 times the first search, and the third was ten times the first search. Blocks estimated using the third search required a minimum of two composites and a maximum of ten composites. The search ellipse was orientated horizontally.

Table 14-7
Search Parameters

Search Distance (m)			Number of Composites		Second Search Multiplier	Number of Composites		Third Search Multiplier	Number of Composites	
X	Y	Z	Min	Max	Factor	Min	Max	Factor	Min	Max
100	100	5	4	12	1.5	4	12	10	2	10

An average density of 1.67t/m^3 was assigned to the block model.

14.9 Model validation

Validation of the grade model was undertaken using the following methods:

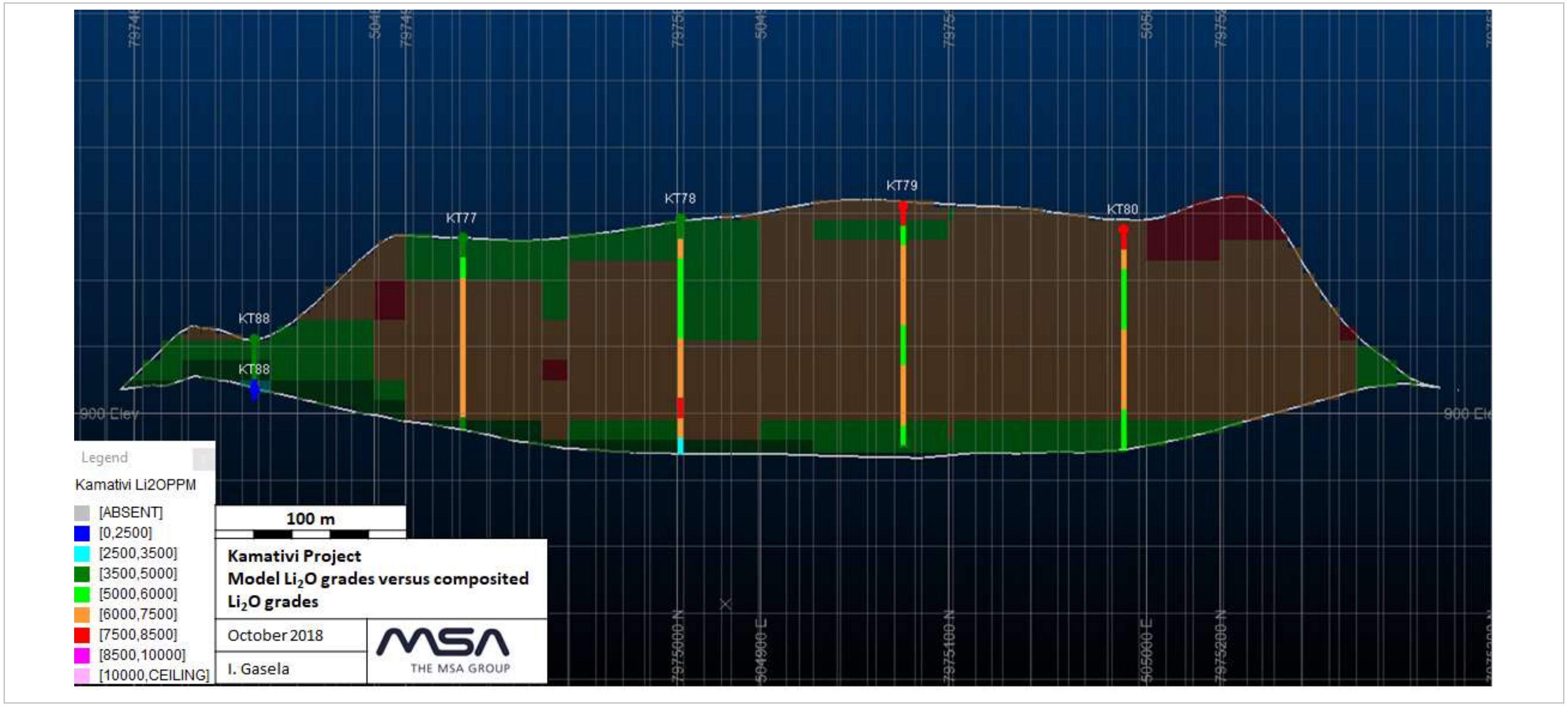
- visual examination of the input data against the block model estimates,
- swath plots, and
- comparison of the input data statistics against the model statistics.

The block model was examined visually in sections to ensure that the drillhole grades were locally well represented by the model. It was found that the model validates reasonably well against the data. A section showing the block model estimates and drill holes grades for Li_2O is presented in Figure 14-10.

Swath plots were prepared that compare the average grades of the block model against the input data along a number of corridors in various directions through the deposit. These show that the Li_2O estimates are representative of the grade trends of the composites across the deposit (Figure 14-11).

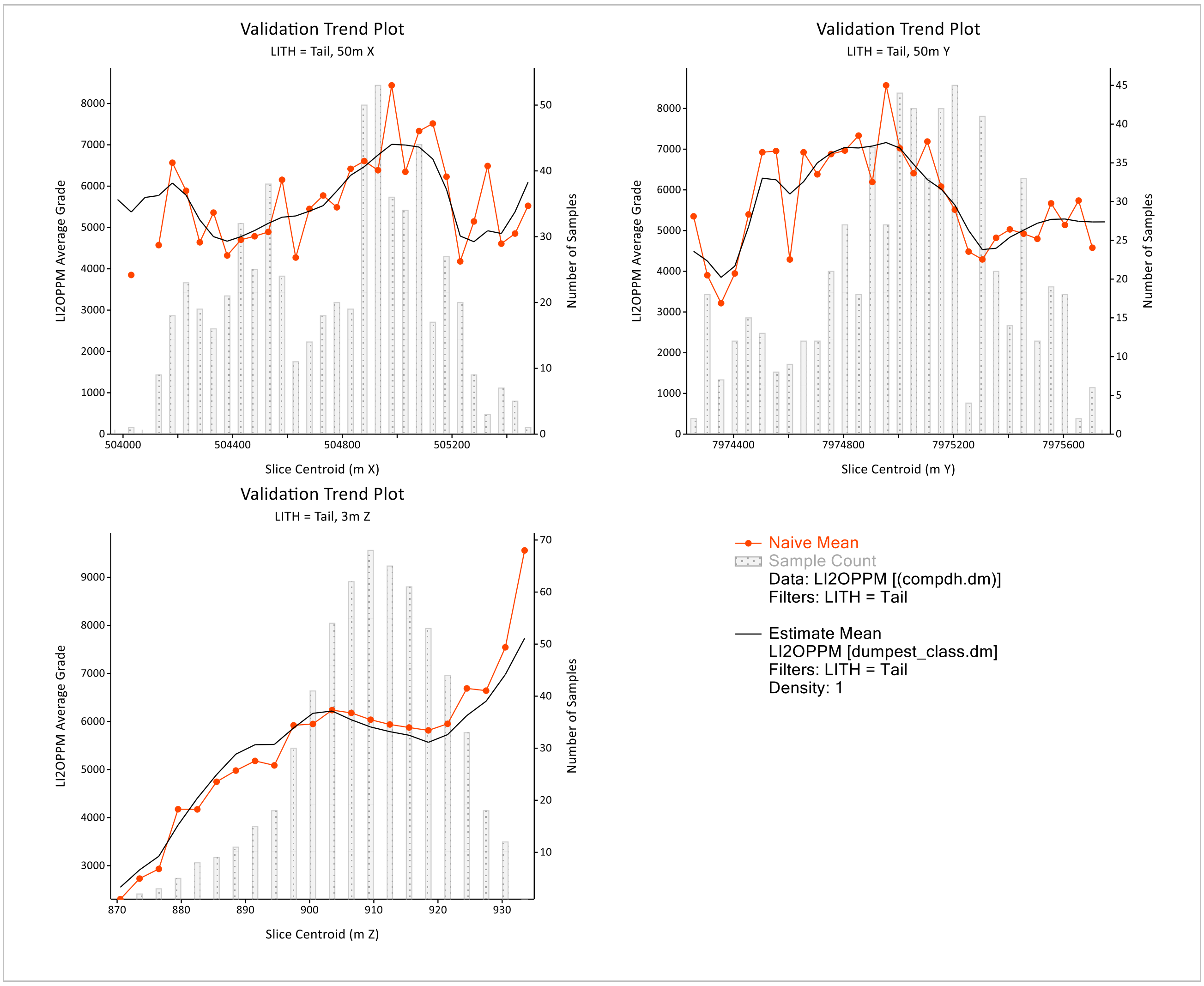
Figure 14-10

North-east cross section showing grade correlation between the model and composite Li_2O grade. Three times vertical exaggeration.



Source: MSA 2018

Figure 14-11
Swath plot for Li_2O grade in the tailings.



Source: MSA 2018

The average grade of the model compares well to the average grade of the composite data (de-clustered to 50 mX by 50 mY by 5 mZ), with no significant relative difference (Table 14-8).

Table 14-8
Comparison between mean model grade and mean input data grade.

	De-clustered mean grade	Block model mean grade	Relative difference (%)
Li_2O (ppm)	5,820	5,822	0.0
SnO_2 (ppm)	494	494	0.0
Ta_2O_5 (ppm)	41	41	-0.4
Fe_2O_3 (%)	1.22	1.22	0.0
Nb_2O_5 (ppm)	65	65	0.0

14.10 Classification

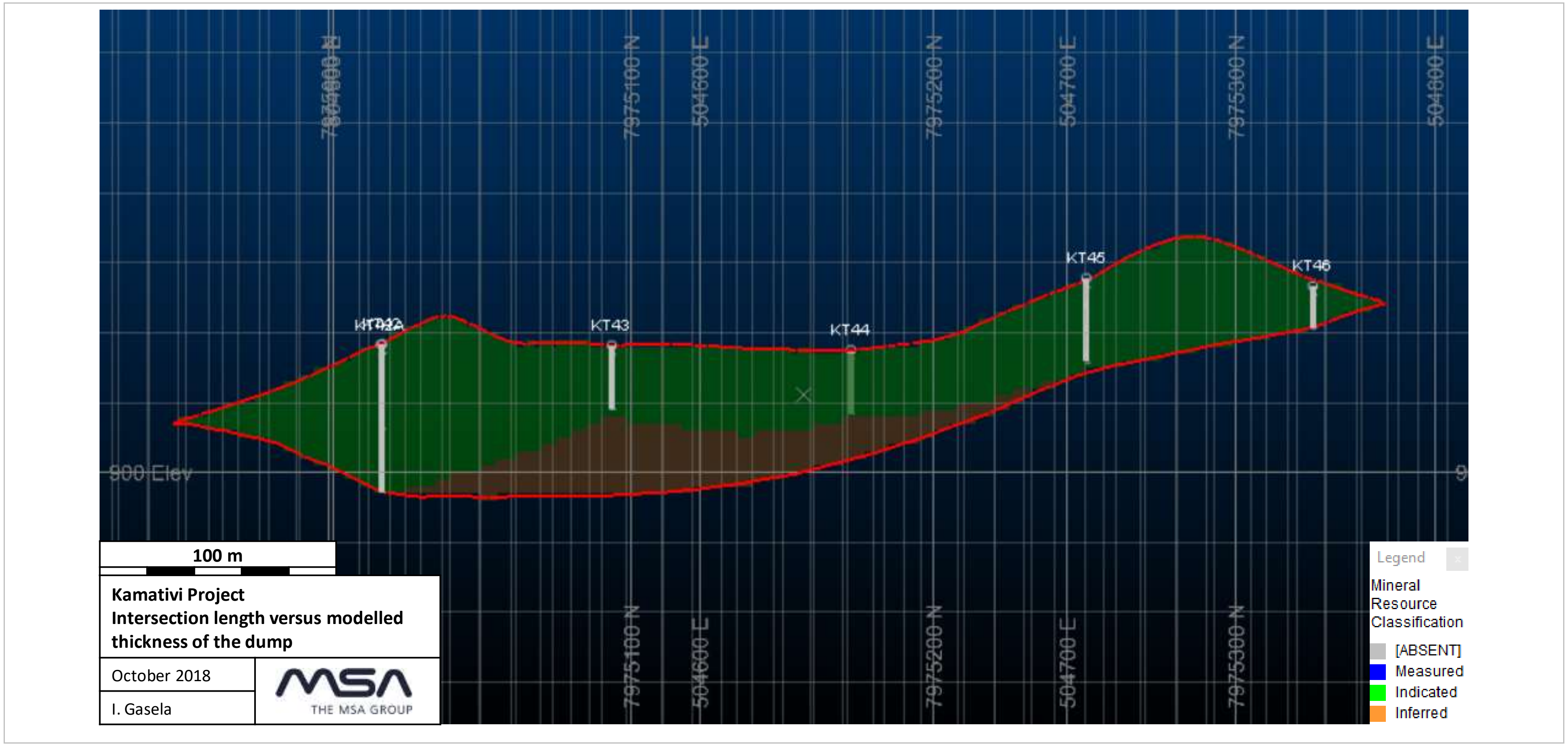
Classification of the Kamativi Tailings Mineral Resource is based on confidence in the data, confidence in the geological model, grade continuity and variability, and the frequency of the drilling data. The main considerations in the classification of the Mineral Resource are as follows:

- The drillhole spacing is sufficient to estimate grades and model the dump framework to a reasonable degree of confidence.
- There is acceptable confidence in the accuracy and integrity of the Kamativi data.
- Two types of drilling were used, generally in separate areas except where twin drilling was undertaken. Comparison of the twinned holes (core and auger) demonstrates that there is no bias in the assay grades from samples obtained with the two methods.
- The surfaces used to define the dump are based on measured data, except for the extrapolated bottom of dump surface. Where drilling continued to the base of the dump the dump model is of high confidence.

The Mineral Resource was classified as Indicated from the top of the dump to the bottom surface of the drill hole intersections and Inferred in an area where the bottom of dump was extrapolated from short holes that did not penetrate the base of the dump over a large area.

Figure 14-12

North-east cross section showing classification of the Kamativi Mineral Resource



Source: *MSA* *2018*

14.11 Mineral Resource Statement

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Resource is classified into the Indicated and Inferred categories as shown in Table 14-9.

The Mineral Resource is reported for the total tailings dump. The lowest Li_2O block estimate is 0.22% which MSA considers has reasonable prospects for eventual economic extraction given the anticipated low-cost bulk mining and non-selective nature of tailings storage facility reclamation, and the outcome of initial metallurgical test work (refer to Item 13 of this Technical Report).

It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and the use of economic parameters used to assess the potential for eventual economic extraction is not an attempt to estimate Mineral Reserves, the level of study so far carried out being insufficient with which to do so.

Table 14-9

Kamativi Lithium Tailings Mineral Resource for the Total Tailings Dump, 10 September 2018

Category	Tonnes (Millions)	Density t/m^3	Li_2O (%)	SnO_2 ppm	Ta_2O_5 ppm	Fe_2O_3 (%)	Nb_2O_5 ppm
Indicated	26.32	1.67	0.58	493	41	1.22	65
Inferred	0.30	1.67	0.62	544	45	1.45	62

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources which are not Mineral Reserves have no demonstrated economic viability.*
3. *Fe_2O_3 is considered a deleterious material and is reported for information purposes.*
4. *Mineralogical work has demonstrated that the lithium mineralogy is dominantly spodumene. Test work indicates that mineralogical separation of the spodumene is possible under laboratory conditions*
5. *For the purposes of establishing reasonable prospects for eventual economic extraction, the cost of producing and delivering one tonne of 6% Li_2O spodumene concentrate, FOB Durban, is estimated at an average of US$286/t based on:*
    - a. *Mining costs of US$26/t*
    - b. *Processing costs of US$110/t*
    - c. *G&A of US$50/t*
    - d. *Transport of US$100/t*
    - e. *The expected revenue derived from one tonne of concentrate is expected to be US$800*

15 MINERAL RESERVE ESTIMATES

Not applicable at this stage.

16 MINING METHODS

Not applicable at this stage.

17 RECOVERY METHODS

Minor work has been done on the recovery methods and a summary of the work done to date is presented below. The reader is referred to Section 13 for more information regarding the test work completed to date.

17.1 Test Work Summary

The deportment of lithium almost exclusively to spodumene lends the project to the improved extraction of lithium-bearing minerals from the Kamativi resource, due to the higher density of spodumene relative to bulk species such as quartz and plagioclase, allowing for beneficiation by means of dense media separation. The major impurity, Fe, is amenable to removal by magnetic separation. An additional consideration is spodumene's amenability to flotation processes. The lithium content of spodumene is also significant relative to the majority of other lithium-bearing minerals.

17.2 Flowsheet Development

On the basis of the test work completed, a likely process flowsheet to produce a concentrate is thus described as follows:

- hydromining of the deposit;
- classification of the material at 400 to 500 µm by screening equipment or cycloning;
- processing of the coarse material fraction in a dense media separation ("DMS") plant
- cleaning of the DMS concentrate by means of wet high intensity magnetic separation ("WHIMS") technology;
- processing of the fine material fraction by means of flotation; and
- cleaning of the flotation concentration by means of WHIMS technology.

18 PROJECT INFRASTRUCTURE

Not applicable at this stage.

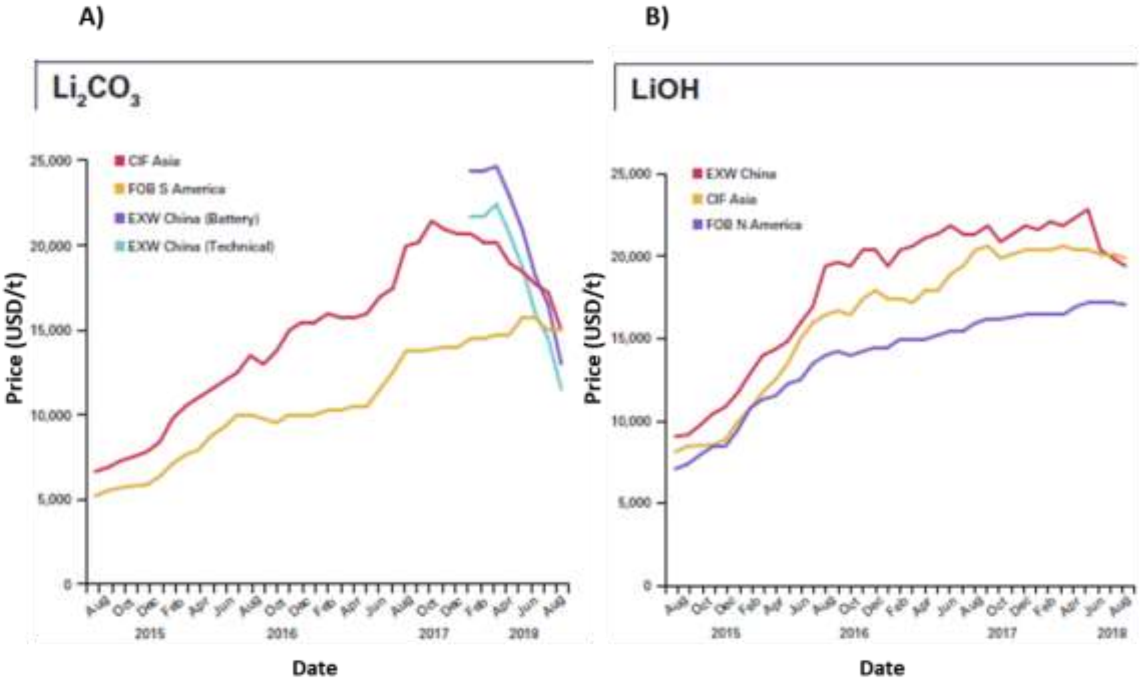
19 MARKET STUDIES AND CONTRACTS

The activity in the lithium market is driven largely by the forecast growth in demand from electric vehicles (EVs), consumer electronics, and growth in the energy storage solutions market (ESS) (McKinsey & Company, 2018) and on the assumption that lithium-ion batteries will be the technology of choice for these markets. This has led to a significant increase in demand, with producers scrambling to secure supply for lithium concentrates for beneficiation into either lithium carbonate and or lithium hydroxide to meet the anticipated demand.

Historically, the majority of lithium production has come from the low-cost brine producers in Chile, China and Argentina. However, new supply from the hard rock operations in Australia, and elsewhere in the world, is competing with the South American brine producers. Many of the lithium brine producers have announced plans or are in the process of expanding production. However, this new supply is only due to come online in the next few years. Production from hard rock operations is anticipated to increase rapidly in the near future, compared with brine producers, and will bring the supply closer to forecasted demand. Currently most of the lithium carbonate and lithium hydroxide producers are located in China. However, there are plans from many of European based automakers to set up conversion plants in Europe make which potentially opens up supply of lithium concentrate from the African market.

Lithium demand is set to double every five years as the overall cost of batteries decreases and the average battery size for vehicles increases (CORMARK, 2018). However, it is forecast that lithium supply will remain tight for 5 – 10 years (CORMARK, 2018). Regulators play a significant role in driving EV penetration in the market, for example the Chinese Government and European Union put ambitious EV adoption targets and strategies in place to combat pollution which will help grow the market (CORMARK, 2018). Benchmark Minerals (2018) suggests that recent fears of oversupply from China leading to a decrease in Lithium price have been myopic and ignore the heterogeneity of lithium feedstocks available. It is important to note the nature of the lithium entering the supply chain; the majority of lithium concentrate being produced in China is not of technical and chemical grade, thus requiring costly up-grading for battery applications. The longer-term nature of contracts for the majority of the market outside of China has, and will continue to, insulate prices in the short-term. Just as the pricing functions built into contracts prevented the "rest of world" prices peaking to China levels in 2017, they will protect prices from falling to China levels in 2018 (Benchmark Minerals, 2018). The recent correction in the lithium carbonate and lithium hydroxide prices (Figure 19-1) saw pricing fall from >$ 20,000/ton to around $ 13,000 and $ 17,000 respectively; with prices still higher than the average of $ 5,000/ton in 2015 (McKinsey & Company, 2018; Benchmark, 2018). By comparison the price of the spodumene feedstock from Australia fell marginally to $ 890/ton.

Figure 19-1
Pricing curves for A) lithium carbonate and B) lithium hydroxide (Benchmark Minerals, 2018)



19.1 Supply of Spodumene Concentrates

Chile, China and Australia have historically been the major producers in the lithium industry, the three countries accounted for 85% of global production (McKinsey & Co, 2018). However, the limited supply capacity from South America and increasing demand have resulted in hard rock lithium deposits becoming viable sources of feedstock. Hard rock producers are forecasted to become the majority source of supply in the near future due to increasing demand and comparative ease of bringing these projects online and of processing compared to brine deposits.

Spodumene concentrate is usually sold as a minimum 6.0 % Li2O concentrate for use in battery manufacturing and other industrial applications (sometimes referred to Talison SC6.0). Iron content is also an important factor for technical grade spodumene concentrates (used in the glass and ceramic industry) with a maximum grade of 1 % Fe_2O_3. The chemical grade spodumene concentrates are generally less strict.

As new mines begin production, the supply of bulk concentrates will increase with grades between 4% - 7% Li_2O. However increased Li_2O production does not necessarily translate immediately into converted carbonate/hydroxide capacity that is required for the manufacturing of battery storage.

Preliminary heavy liquid separation test work on the +425 µm fraction of material from the Kamativi tailings which contains 55 % of the Li_2O, has demonstrated that a concentrate containing ~6.1% Li_2O and 0.93 % Fe (or 1.2 % Fe_2O_3), which equates to a 74 % Li_2O and 20 % Fe recovery, is possible. Magnetic separation of the Fe from this concentrate was able to produce a final concentrate assay of 7.08% Li_2O and 0.26% Fe. Further floatation test work on the -425 µm fraction is ongoing as is dense media separation pilot scale test work.

The test work to date has demonstrated that the production of a marketable spodumene concentrate is possible.

19.2 Contracts

Two term sheets have been received from international commodity traders expressing interest in the off-take of the concentrate produced from the Project. The exact pricing mechanism has yet to be determined between the parties however it has been proposed that the material will be sold on an FCA Kamativi basis with a price review and adjustment every 6 months. The anticipated concentrate grade is 6 % Li_2O but will be finalised upon completion of test work. As per confidentiality agreements in place between all parties involved in the ongoing off-take negotiations, the names of the commodity traders cannot be disclosed.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

Not applicable at this stage.

20.2 Permitting

No permitting is required to undertake the next phase of auger drilling on the tailings material.

20.3 Social or Community Impact

It is anticipated that The Project will have a positive social impact on the village of Kamativi and the surrounding areas. It is anticipated that should the project advance, social upliftment will be achieved through the employment of many of the local population, in addition to other social programmes such as the upgrading of village amenities and infrastructure.

21 CAPITAL AND OPERATING COSTS

Not applicable at this stage.

22 ECONOMIC ANALYSIS

Not applicable at this stage.

23 OTHER RELEVANT DATA AND INFORMATION

Not applicable

24 ADJACENT PROPERTIES

The Property is located within a larger licence, ML No. 12 that includes the old Kamativi Tin Mine which is held by the ZMDC.

The rights to the hard rock lithium hosted pegmatite mineralisation within ML No. 12 are currently held in a JV with China Beijing Pinchang. The history of the mining activity is summarised in Section 6 of this report.

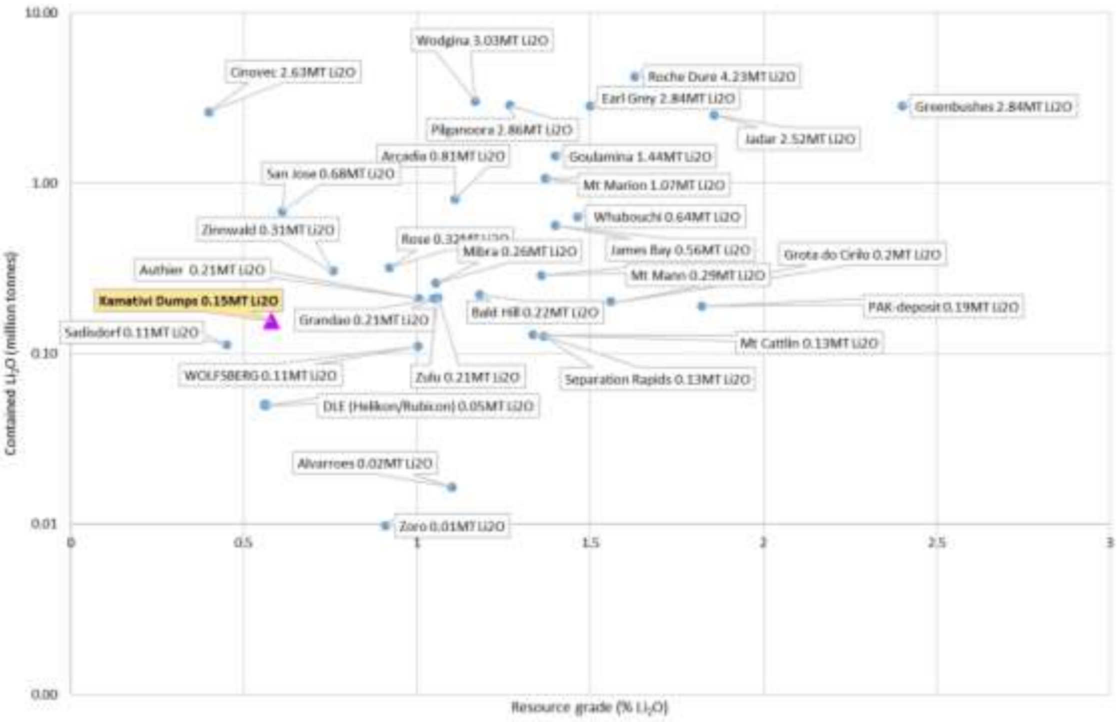
25 INTERPRETATION AND CONCLUSIONS

The Kamativi project is located approximately 310 km northwest of Bulawayo and comprises the lithium-bearing tailings dump derived from the mining of tin at the Kamativi Tin Mine. The mine operated for a period of approximately 60 years, from 1936-1994 and focused on extraction of cassiterite from the tin-bearing LCT pegmatites on the property. None of the lithium-bearing minerals were recovered. Spodumene is the dominant lithium mineral in the pegmatite and thus also in the tailings material. Other lithium minerals identified include cookeite, zinnwaldite, petalite and amblygonite.

The Kamativi Lithium Tailings Project is compared other hardrock pegmatite hosted lithium projects globally in Figure 25-1.

Figure 25-1

Comparison of the Kamativi Project with selected global pegmatite-hosted lithium projects



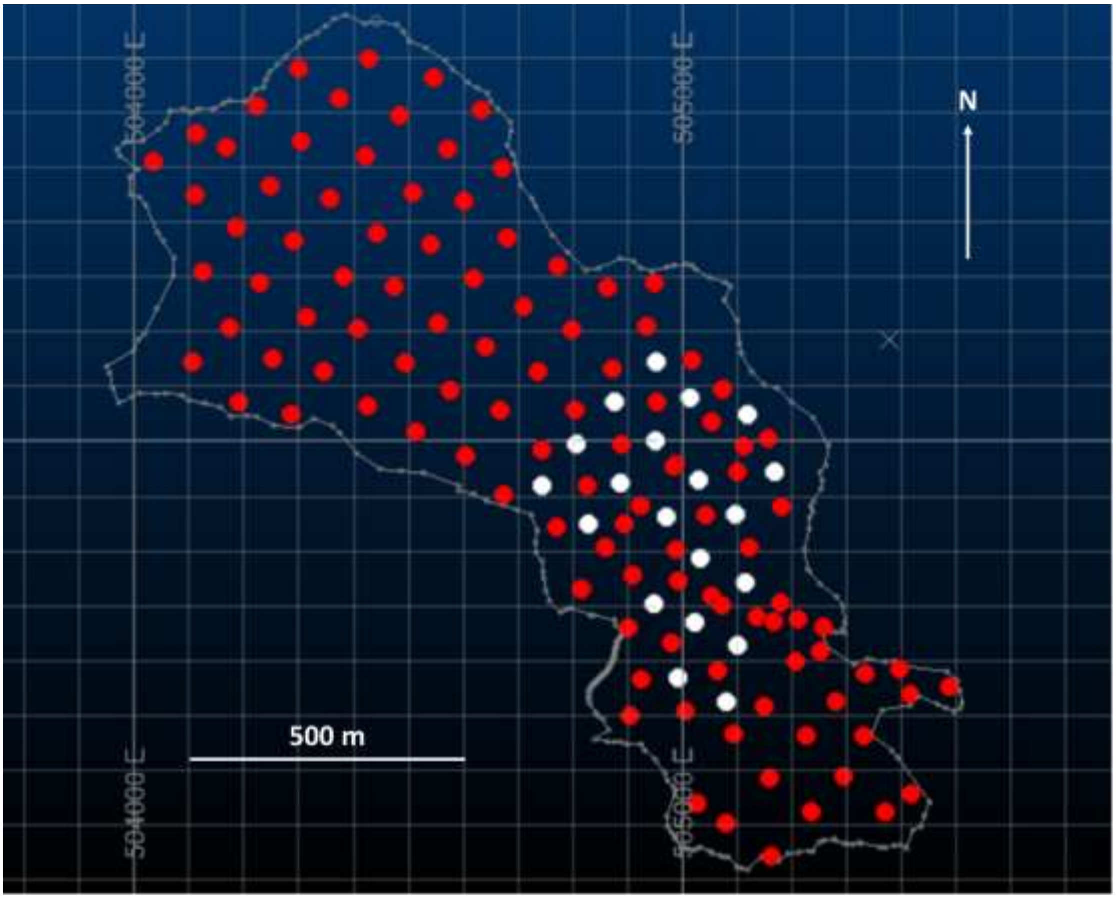
Preliminary metallurgical test work suggests spodumene is recoverable through a combination of gravity and flotation methods and that a potentially saleable spodumene concentrate can be produced. However more detailed work is required in order to establish the most efficient work flow and also a suitable method for the removal of iron from the concentrate.

26 RECOMMENDATIONS

The QPs recommend the following work programme going forward (see also Table 26-1):

- Complete a Preliminary Economic Assessment (PEA) in order to improve the understanding of the economic merits of the Project.
  - Estimated cost of $80,000.
- Should the PEA be encouraging, conduct further exploration to improve confidence in the Mineral Resource in order to upgrade the higher grade area of the Mineral Resource into the Measured category:
  - Infill drilling to complete a 50 m grid over the the high grade area (Figure 26-1)
  - It is estimated that a further 20 holes drilled using an auger will be required, with an average length of 25 m. At an all-in cost of $110 per metre, with an estimated cost of $55,000.

Figure 26-1
Comparison of the Kamativi Project with selected global pegmatite hosted lithium projects.



- Additional metallurgical test work taking into account possible variations in grade and mineralogy identified in the drilling programme and optimise the gravity process, flotation, grind size, and removal of the iron from the final concentrate. The test work should be conducted on a large representative sample from the Kamativi tailings dump. This test work, with the relevant data being used as inputs for flowsheet design, will be carried out according to the following individual test campaigns:
    - dense media separation pilot test work;
    - magnetic separation test work; and
    - flotation test work.
- Dense media separation pilot scale test work should be conducted on a coarser fraction of the tailings material to ascertain the feasibility of the production of a spodumene pre-concentrate through the application of dense media separation technology. Magnetic separation test work will likely be required to remove iron-bearing species from the HLS spodumene concentrate. Test work will involve the processing of concentrate material at various magnetic intensities, for the purposes of determining optimum operating parameters for the magnetic separation unit.
- Due to the low efficiencies realised during the processing of fine material in a DMS circuit, flotation test work should be completed on the finer fraction of the tailings material. Optimum operating conditions and reagent addition will be determined, in addition to the number of processing stages required. Grind optimisation and lock cycle flotation will also be conducted. Depending on the level of Fe rejection achieved during the flotation test work, magnetic Fe removal work may be required on the flotation concentrate.
- Finalisation of the location of the new tailings disposal facility and proposed process plant. This will need to be done in conjunction with the Environmental Impact Assessment.

Table 26-1

Summary of proposed exploration programme for next phase of exploration

Items	Key Quantities	Budget (USD)	Proposed Deliverables
Auger drilling programme	Approximately 525 m drilling and assays, Mineral Resource estimate and geological consulting	55,000	20 drill holes averaging 25m depth. Mineral Resource estimate brought to code-compliant category
PEA		80,000	
Advanced metallurgical test work	Flotation, grind size determination, magnetic separation and gravity test work	150,000	Metallurgical process work flow
Environmental Impact Assessment	Hydrological study, hydrogeology, soils, water, biological field studies, tailings and pilot plant	100,000	Initiation of EIA Process

	processing permitting.		

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APPENDIX 1: Acronyms and Abbreviations

Acronyms and Abbreviations

AMIS	African Mineral Standards
Ave.	Average
Be	Beryllium
BSE	Back Scatter Electron
CIF	Cost, insurance and freight
Chimata / the Company	Chimata Gold Corporation
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Columbo-tantalite or Coltan	Coltan (short for columbite–tantalite and known industrially as tantalite) is a dull black metallic ore from which the elements niobium and tantalum are extracted. The niobium-dominant mineral in coltan is columbite (Nb_2O_5) and the tantalum-dominant mineral is tantalite (Ta_2O_5).
CP	Competent Person
CRM	Certified Reference Material
Cs	Caesium
DD	Diamond drilling
DMS	Dense Media Separation
DTM	Digital terrain model
EA	Environmental Assessment
ECC	Environmental Clearance Certificate
EIA	Environmental Impact Assessment
EM	Electromagnetic
EMA	Environmental Management Act
EMP	Environmental Management Plan
EPL	Exclusive Prospecting Licence
EU	European Union
FCA	"Free Carrier" means that the seller fulfils his obligation to deliver when he has handed over the goods, cleared for export, into the charge of the carrier named by the buyer at the named place or point
FCTR	Flotation characterisation test rig
Fe	Iron
GPS	Global positioning system - An instrument used to locate or navigate, which relies on three or more satellites of known position to identify the operators location.
GSSA	Geological Society of South Africa
HLS	Heavy liquid separation
ICP	Inductively coupled plasma
IDC	Industrial Development Corporation of Zimbabwe
Jimbata	Jimbata (Pvt) Ltd – to be renamed The Lithium Tailings Company of Zimbabwe
JV	Joint Venture
Kamativi tailings / Kamativi	Kamativi Lithium Tailings Project

Project	
LCE	Lithium carbonate equivalent. Conversion from %Li_2O to %LCE = Li_2O(%)*2.473 %Li to %LCE – Li(%)*5.323
LCT	Lithium Caesium Tantalite
Li	Lithium
Li_2O	Lithium oxide
Li_2O (Lithia)	Lithia - Lithium oxide (Li_2O). Contains 46.4% Li. Conversion from %Li to %Li_2O = %Li*2.153
Li_2CO_3	Lithium carbonate - Tradable lithium concentrate used for Li-ion battery manufacture. Contains 18.8 % Li
$LiOH.H_2O$	Lithium hydroxide monohydrate - Tradable lithium concentrate used for Li-ion battery manufacture. Contains 16.5 % Li
Lintmar	Lintmar (Private) Limited
LOI	Loss on ignition
Ma	Million years.
Nb-Ta	Columbo-tantalit/ coltan
NORM	Naturally occurring radioactive materials
NYF	Niobium Yttrium Flourine
mamsl	Metres above mean sea level
MGS	Multi gravity separator
ML	Mining Lease
MSA	The MSA Group (Pty) Ltd
Monazite	Phosphate mineral with a chemical composition of $(Ce,La,Nd,Th)(PO_4,SiO_4)$. It usually occurs in small isolated grains, as an accessory mineral in igneous and metamorphic rocks such as granite, pegmatite, schist, and gneiss.
MT	Magneto-Telluric
Mt	Million tonnes
Nb	Niobium
nCZ	northern Central Zone
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects
NUST	New University of Science and Technology
N.V.	Naamloze Vennootschap / public company (Dutch)
QP	Qualified Person(s)
ppm	Parts per million
PSD	Particle size distribution
QA/QC	Quality Assurance and Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
Rb	Rubidium
RC	Reverse Circulation (drilling)
Report	Independent Technical Report

RQD	Rock Quality Designation
SACNASP	South African Council for Natural Scientific Professions
SANAS	South African National Accreditation System
sCZ	Southern Central Zone
SD	Standard Deviation
SEIA	Social and Environmental Impact Assessment
SG	Specific Gravity
SGS Randfontein	SGS South Africa (Pty) Ltd Laboratory, Randfontein, South Africa
SnO_2	Tin oxide / Cassiterite
STI	Sexually transmitted infections
t	Metric tonnes
Ta	Tantalum
TBE	tetrabromoethane
the Project	Kamativi Lithium Tailings Project
the Property	Kamativi Lithium Tailings Property
tpm	Metric tonnes per month
tpd	Metric tonnes per day
UNIDO	United Nations Industrial Development Organization
US$ / USD	United States of America dollars
WHIMS	Wet high intensity magnetic separation
XRD	X-ray powder diffraction
XRF	X-Ray Fluorescence
ZIM	Zimbabwe Lithium Company (Mauritius) Limited
ZMDC	Zimbabwe Mining Development Corporation

APPENDIX 2: Table of drill hole information and collar positions

BHID	Coordinates - Arc 1950 (Zambia) UTM 35K			Drill Hole Type	End of Hole	Comments
	X	Y	Z			
KT01	505415.05	7974538.33	925.36	DD	3.90	
KT02	505332.97	7974576.04	922.53	DD	3.30	
KT03	505251.26	7974616.73	918.85	DD	3.20	
KT04	505257.32	7974660.81	920.38	DD	4.30	
KT05	505165.97	7974671.18	917.57	DD	8.60	
KT06	505071.46	7974701.05	916.48	DD	10.79	
KT07	504992.18	7974745.70	911.50	DD	18.50	
KT08	504908.94	7974756.02	910.97	DD	20.80	
KT09	505178.66	7974705.70	919.72	DD	6.00	
KT10	505210.55	7974675.37	917.46	DD	10.50	
KT11	505486.88	7974551.98	925.62	Auger	2.96	
KT12	505395.79	7974584.54	923.20	Auger	2.96	
KT13	504190.18	7975071.60	900.68	Auger	4.80	
KT14	504252.45	7975151.05	914.83	Auger	21.00	
KT15	504314.01	7975226.67	920.79	Auger	28.93	
KT16	504381.43	7975300.89	921.73	Auger	25.46	
KT17	504442.57	7975380.10	923.00	Auger	25.25	
KT18	504507.98	7975454.17	927.37	Auger	28.30	
KT19	504571.23	7975534.02	924.54	Auger	19.90	
KT20	504632.67	7975605.23	911.25	DD	5.50	
KT22	504285.79	7975049.56	905.58	Auger	5.30	
KT23	504345.47	7975128.21	918.74	Auger	18.40	
KT24	504407.59	7975205.59	919.85	Auger	20.86	
KT25	504473.88	7975282.78	920.79	Auger	25.45	
KT26	504540.10	7975359.77	924.30	Auger	25.80	
KT27	504601.09	7975438.59	932.40	Auger	24.68	
KT28	504671.07	7975498.95	918.73	DD	5.50	
KT32	504425.38	7975064.56	917.33	Auger	16.30	
KT33	504493.93	7975143.35	919.27	Auger	22.50	
KT34	504553.79	7975215.42	919.36	Auger	22.40	
KT35	504618.75	7975297.33	922.03	Auger	19.40	
KT36	504680.18	7975372.01	930.38	Auger	14.90	
KT42	504513.53	7975017.50	917.61	DD	12.00	
KT42A	504514.09	7975016.40	917.55	Auger	21.75	
KT43	504576.03	7975093.37	917.40	Auger	10.04	
KT44	504640.69	7975172.84	916.76	Auger	9.80	
KT45	504710.30	7975246.22	927.04	Auger	13.26	
KT46	504772.46	7975320.62	925.91	DD	6.50	
KT53	504603.98	7974973.69	916.76	Auger	18.00	
KT54	504667.01	7975056.80	912.97	Auger	18.00	
KT55	504736.19	7975127.67	920.31	Auger	19.50	

KT56	504797.70	7975204.31	931.61	Auger	16.50	
KT57	504863.21	7975281.03	916.95	DD	8.00	
KT64	504673.47	7974902.43	910.16	DD	30.90	
KT65	504743.21	7974984.90	919.09	Auger	22.50	
KT66	504803.84	7975057.22	923.85	Auger	30.90	
KT67	504871.79	7975132.84	931.63	Auger	28.50	
KT68	504934.43	7975210.51	920.63	Auger	24.90	
KT69	504948.69	7975289.38	910.11	DD	11.30	
KT77	504825.52	7974919.17	925.62	Auger	29.00	
KT78	504888.51	7974994.76	928.49	Auger	35.20	
KT79	504953.12	7975071.72	930.37	Auger	37.30	
KT80	505016.52	7975148.46	926.88	Auger	33.20	
KT88	504769.83	7974843.60	910.38	DD	9.00	
KT89	504814.78	7974730.59	912.87	DD	10.50	
KT89A	504817.56	7974730.31	912.81	Auger	15.00	
KT90	504900.73	7974660.12	892.22	DD	2.40	Not Sampled-Drilled into Base of Dump
KT91	504979.80	7974632.38	894.88	DD	13.09	
KT92	505064.05	7974581.39	898.97	DD	13.50	
KT93	505148.70	7974516.20	902.16	DD	6.00	
KT94	505225.64	7974463.17	899.88	DD	22.00	
KT95	505293.54	7974388.53	900.39	DD	14.80	
KT96	505369.72	7974323.87	904.39	DD	17.65	
KT97	504858.83	7974806.99	913.13	DD	16.50	
KT97A	504861.27	7974806.48	912.81	Auger	21.55	
KT98	504922.87	7974882.24	926.18	DD	18.90	
KT98A	504893.39	7974849.45	925.67	Auger	21.70	
KT98B	504923.64	7974883.68	925.96	Auger	36.00	
KT99	504983.80	7974954.34	930.71	DD	15.00	
KT99A	504986.64	7974959.23	931.57	Auger	31.50	
KT100	505053.05	7975036.10	930.56	DD	17.00	
KT101	505072.59	7975094.89	931.97	DD	24.00	
KT101A	505073.96	7975097.18	931.79	Auger	28.50	
KT105	504987.84	7974802.60	925.22	DD	24.40	
KT106	505042.12	7974865.65	926.75	DD	23.80	
KT107	505100.30	7974944.45	930.16	DD	17.40	
KT108	505155.79	7975005.20	922.96	Auger	14.95	
KT108A	505112.29	7974991.20	933.41	Auger	23.46	
KT112	505052.85	7974717.90	916.33	DD	18.00	
KT113	505121.39	7974806.17	927.84	DD	18.31	
KT114	505180.19	7974881.01	925.67	Auger	24.00	
KT119	505135.63	7974679.55	916.50	DD	6.00	
KT125	505206.18	7974598.51	915.61	DD	5.00	
KT130	505279.57	7974525.31	906.70	DD	8.00	
KT136	505330.43	7974462.20	903.09	DD	5.05	
KT143	505417.00	7974356.52	901.28	DD	12.65	
KT155	504924.59	7974565.53	892.85	DD	14.00	

KT156	505004.71	7974507.71	898.29	DD	10.00	
KT158A	505158.59	7974385.42	894.09	DD	21.40	
KT157	505093.16	7974465.71	901.19	DD	17.60	
KT159	505234.65	7974324.21	899.14	DD	19.50	
KT164	504112.96	7975561.86	899.44	DD	9.00	
KT165	504110.79	7975450.14	907.41	DD	6.00	
KT166	504106.13	7975145.32	899.06	DD	12.00	
KT167	504167.81	7975537.38	907.07	DD	8.72	
KT168	504223.41	7975613.10	904.25	DD	3.00	
KT169	504299.62	7975680.64	910.68	DD	9.00	
KT170	504034.68	7975511.14	896.75	Auger	2.96	
KT200	504125.04	7975309.65	914.44	Auger	4.28	
KT201	504186.03	7975390.97	920.74	DD	16.50	
KT202	504247.46	7975466.34	922.66	Auger	18.80	
KT203	504304.04	7975547.63	923.01	Auger	20.90	
KT204	504373.58	7975626.73	921.70	DD	13.50	
KT205	504427.53	7975698.97	912.97	DD	9.00	
KT206	504173.88	7975207.77	913.19	Auger	24.48	
KT207	504228.84	7975289.53	921.76	Auger	27.70	
KT208	504290.85	7975365.97	923.55	Auger	20.34	
KT209	504356.65	7975443.64	924.18	Auger	25.30	
KT210	504421.82	7975520.84	926.36	Auger	28.50	
KT211	504483.54	7975595.17	929.27	Auger	28.50	
KT212	504545.84	7975664.17	910.29	DD	6.00	
KT213	504904.17	7974499.51	893.65	DD	15.60	
KT214	505026.73	7974338.76	880.87	DD	8.70	
KT215	505079.23	7974303.21	883.26	DD	15.00	
KT216	505161.54	7974242.38	886.80	DD	6.40	