Specialist Consultants to the Mining Industry

Tantalex Lithium Resources Corp.
Manono Lithium Tailings Project
Democratic Republic of Congo

NI 43-101 Technical Report – 04 September 2023 Mineral Resource Estimate

Prepared By The MSA Group (Pty) Ltd for:
Tantalex Lithium Resources Corp.

Prepared by:
Rui Goncalves Pr. Sci. Nat

Effective Date: 23 August 2023
Report Date: 04 September 2023

MSA Project No.: J4587

IMPORTANT NOTICE

This report was prepared as a National Instrument NI 43-101 Technical Report for Tantalex Lithium Resources Corp (Tantalex) 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 Tantalex 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.

CERTIFICATE OF QUALIFIED PERSON

I, Rui Goncalves, Pr.Sci.Nat., do hereby certify that:

1. I am a Senior Mineral Resource Consultant of:

   The MSA Group (Pty) Ltd
   Henley House, Greenacres Office Park,
   Victory Park, Randburg, 2195
   South Africa

2. This certificate applies to the technical report titled "Tantalex Lithium Resources Corp., Manono Lithium Tailings Project, Democratic Republic of Congo, NI 43-101 Technical Report – 04 September 2023", that has an effective date of 23 August 2023 and a report date of 04 September 2023(the Technical Report).

3. I graduated with a BSc (Hons) degree in Geology from the University of Pretoria in 2010. In addition, I obtained a Master of Science degree in Engineering from the University of Witwatersrand in 2021.

4. I am a registered Professional Natural Scientist (Geological Science) with the South African Council for Natural Scientific Professions (SACNASP) and a Member of the Geological Society of South Africa.

5. I have worked as a geologist for a total of 13 years, during which time I have worked in a number of roles in precious and base metal exploration, mine geology and Mineral Resource estimation. I have conducted Mineral Resource estimates and reviews for a wide range of commodities and styles of mineralisation including copper-cobalt, gold, tin, nickel, platinum group elements, rare earth elements and niobium. Specific tailings experience includes copper-cobalt deposits in the Democratic Republic of Congo (DRC), tin experience includes shear-hosted tin deposits in the DRC.

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 Manono Tailings property for two days from 29 to 30 April 2022.

8. I am responsible for the preparation of items 1 to 12 and 14 to 24.

9. I have not had prior involvement with the property that is 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 04th day of September, 2023.

"signed and stamped"

(Rui Goncalves, Pr. Sci. Nat)

Statement of Certification by Author

I, Antoine Lefaivre, P.Eng., as an author of the Technical Report titled "*Tantalex Lithium Resources Corp., Manono Lithium Tailings Project, Democratic Republic of Congo, NI43-101 Technical Report –04 September 2023*" effective date: August 23rd, 2023, do hereby certify that:

1. I am a Lead Process Engineer at Novopro Projects Inc., 1350 Sherbrooke West, Suite 600, Montreal QC, H3G 1J1, Canada.
2. I am a graduate of Ecole Polytechnique, Montreal, Quebec, Canada with a B.Sc. Chemical Engineering 2007.
3. I am a member in good standing of the Ordre des Ingénieurs du Québec, license no. 5002027.
4. I have over 15 years of experience executing industrial projects, economic and feasibility studies, process development, and due-diligence reviews, and have participated in projects for potash, lithium, magnesium products, using both conventional and solution mining for ore recovery in Canada, United States, Africa, South America and Australia.
5. 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 meet the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am co-author of the report titled "Tantalex Lithium Resources Corp., Manono Lithium Tailings Project, Democratic Republic of Congo, NI43-101 Technical Report – 04 September 2023" effective date: August 23rd, 2023, being author for Item 13.
7. I did not undertake a site visit because of COVID restrictions and the ongoing conflict in DRC.
8. I have not had any prior involvement with the property previous to this Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of the Technical Report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with Tantalex Lithium Resources Corporation.
11. I am independent of Tantalex Lithium Resources Corporation as defined by Section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated the 04th day of September 2023

(signed and sealed)

____

Antoine Lefaivre, P.Eng. (Quebec)

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1 SUMMARY

The MSA Group (Pty) Ltd (MSA) was commissioned by Tantalex Lithium Resources Corp (Tantalex) to complete a Mineral Resource Estimate for the Manono Lithium Tailings Project (Manono or the Project) that is documented in an Independent Technical Report as per National Instrument 43-101 "Standards of Disclosure for Mineral Projects". Manono is a lithium-tin-tantalum tailings project located in the Tanganyika Province of the Democratic Republic of Congo.

1.1 Property Description and Ownership

Tantalex Lithium Corp. (Tantalex) is a Canadian exploration company listed on the Canadian Securities Exchange, the Frankfurt Stock Exchange and the United States OTCQB Venture Market. The subject of this report is the Manono lithium-tin-tantalum tailings deposit, located 490 km north of Lubumbashi, in the Tanganyika Province of the Democratic Republic of Congo (DRC).

The Manono tailings are located within the Tailings Exploitation Permit PER 13698, which is located adjacent to the town of Manono. It consists of 11 tailings dumps spanning a length of 12 km from the southwest towards the northeast. The license is held by Minocom Mining SAS, of which Tantalex holds 52%; 18% is held by MINOR and the remaining 30% by COMINIERE.

1.2 Geology and Mineralisation

The Manono tailings are technogenic deposits, created from the processing of material from the Manono-Kitolo deposit, which was mined from 1919 to the mid-1980's for tin and columbite-tantalite (coltan). Nine out of the eleven tailings were drilled, of which five form this Mineral Resource Estimate. The tailings deposits stretch over a length of 12 km, in a northeast-southwest direction, immediately adjacent to the mined pits. Several of the deposits consist of a mixture of material types, typically pegmatite and laterite, with some clay material being present in minor quantities in specific deposits.

The deposits are named alphabetically, with a suffix used to differentiate between coarse (c) and fine (f) material. The nine tailings that make up the project are from north to south named Cc, Cf, Ec, Hc, Hf, Gc, Gf, Ic and K.

The lithium mineralisation is primarily hosted in spodumene with minor lepidolite. Tin mineralisation is hosted in cassiterite and tantalum in tantalite.

1.3 Exploration Status

The nine tailings deposits have been evaluated by aircore drilling, completed from September 2021 to July 2022. A total of 368 drillholes, amounting to 11 922.4 metres of drilling, have been completed, which took place over two phases.

Drilling was orientated vertically, with the densest drilling found on the K deposit, where holes were spaced 40 m apart. The Gf and Hf deposits were drilled at a spacing of 80 m. The remaining deposits were drilled on an irregular spacing ranging from 20 m to 80 m. Most of the drilling has intercepted the contact representing the pre-depositional surface.

1.4 Mineral Resource Estimate

The Manono tailings were visited by Rui Goncalves, who is a Senior Mineral Resource Geologist with The MSA Group (Pty) Ltd (MSA) and the Qualified Person for this Mineral Resource estimate, on 29 and 30 April 2022. The occurrences and setting of the lithium mineralisation were observed in the field as well as in a selection of chip samples from the first phase of drilling. No drilling was taking place at the time of the site visit, however discussions with Tantalex and observations on-site indicated that reasonable documented procedures and protocols were used in the drilling.

The assay results received from the primary laboratory (SGS in Johannesburg, South Africa) were subjected to a quality assurance and quality control programme and the assays have been confirmed by check assays completed by ALS (Ireland). Both these laboratories are commercial laboratories independent of Tantalex and MSA.

The drilling, logging, sampling and assay data is contained in Microsoft Excel spreadsheets, which were validated by MSA prior to use in Mineral Resource estimation.

Three dimensional volumes were constructed for each tailings deposit. Where applicable, individual volumes representing pegmatite, laterite and clay layers were modelled for each deposit.

Ordinary Kriging was used to estimate lithium oxide (Li_2O), tin (Sn) and tantalum (Ta) grades into a three-dimensional block model for the K deposit. Due to the paucity of the data, inverse distance squared was used to estimate the grades for the remaining seven deposits. Tin and tantalum was only estimated for the K, Gf, Gc and Ic deposits. One deposit (Cf) was not estimated at all due to insufficient drilling coverage.

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines (2019) 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 Resources were classified into the Measured, Indicated and Inferred categories for each deposit and reported at a cut-off grade of 0.20% Li_2O (Table 1-1). The cut-off grade was calculated based on a mining cost of 2.17 USD/tonne, a processing cost of 11.18 USD/tonne, transport cost of 361 USD/tonne, G&A costs of 76.5 USD/tonne, marketing costs of 178.4 USD/tonne, a mining recovery of 99%, process recovery of 63% and a lithium price of 2800 USD/tonne for spodumene concentrate (SC6), which the QP considers will satisfy "reasonable prospects for eventual economic extraction". No Mineral Resources for the Ec, Hc and Hf deposits were declared.

Table 1-1
Manono Mineral Resources a 0.20% Li_2O cut-off grade – 23 August 2023

Deposit	Classification	Tonnes (Mt)	Li_2O %	Sn ppm	Ta ppm
Cc	Inferred	2.99	0.32	-	-
Ic	Inferred	0.51	0.49	583	29
Gc	Indicated	0.29	0.78	579	30
	Inferred	0.51	0.84	554	29
Gf	Indicated	1.39	0.35	183	22
	Inferred	0.13	0.33	209	26
K	Measured	3.77	0.86	305	25
	Inferred	2.33	0.67	652	35
Li_2O, Sn and Ta Mineral Resources					
Total	Measured	3.77	0.86	306	25
	Indicated	1.69	0.42	252	24
	Measured & Indicated	**5.46**	**0.73**	**289**	**25**
	Inferred	3.48	0.66	614	33
Li_2O only Mineral Resources					
Total	Inferred	2.99	0.32	-	-

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources are not Mineral Reserves, have no demonstrated economic viability*
3. *Li_2O % grades calculated by applying a factor of 2.153 to Li % grades*
4. *Mt = Million tonnes, ppm = parts per million*
5. *Inferred Li_2O, Sn and Ta Mineral Resources are totalled for the Southern Sector dumps (Ic, Gc, Gf and K).*
6. *Inferred Li_2O only Mineral Resources are for the Cc dump.*

1.5 Conclusions and Recommendations

The Mineral Resources presented in this Technical Report represent an update to the Mineral Resource estimate with an effective date 13 December 2022 and now includes tin and tantalum.

Additional drilling is recommended for several deposits in order to improve the confidence in the Mineral Resource estimates.

2 INTRODUCTION

The MSA Group (Pty) Ltd (MSA) was commissioned by Tantalex Lithium Resources Corp (Tantalex) to complete a Mineral Resource Estimate for the Manono Lithium Tailings Project (Manono or the Project) that is documented in an Independent Technical Report as per National Instrument 43-101 "Standards of Disclosure for Mineral Projects". Manono is a lithium-tin-tantalum tailings project located in the Tanganyika Province of the Democratic Republic of Congo.

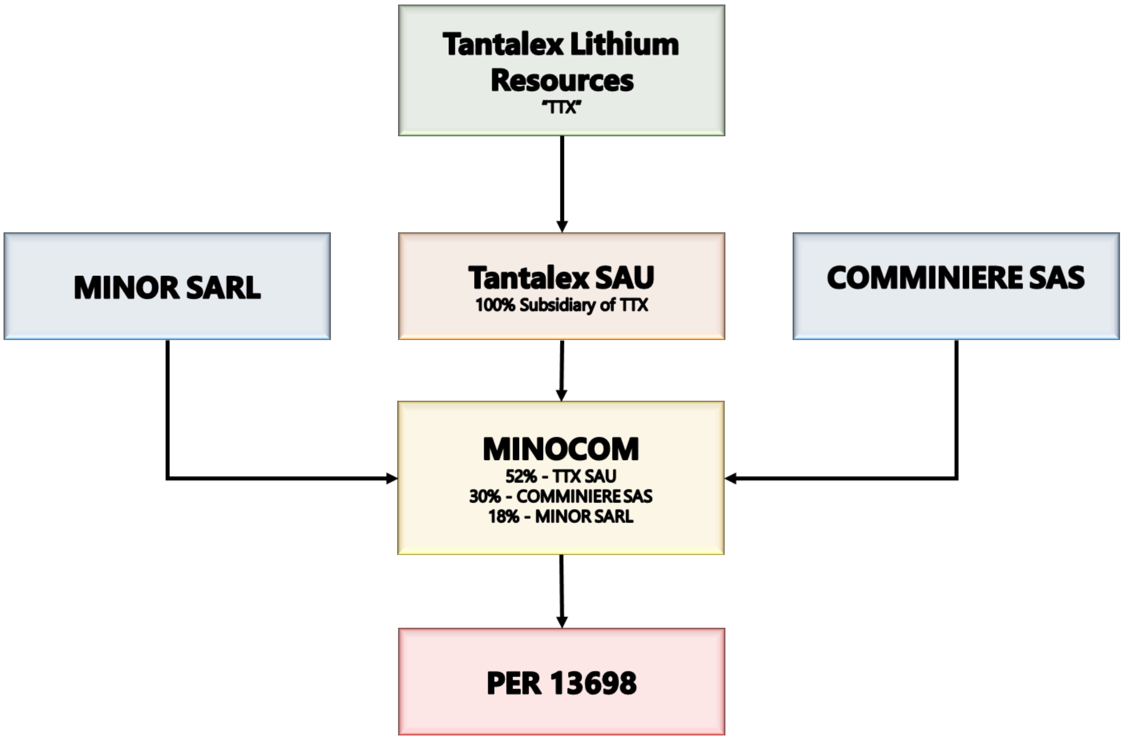
The mineralisation is contained in technogenic deposits, formed from the processing of lithium-caesium-tantalum (LCT) pegmatites of the historical Manono-Kitotolo (MK) mine which operated from 1919 to the mid-1980's. During this time, the mine produced an estimated 140 000 to 180 000 tonnes of tin and 4 500 tonnes of coltan (columbite-tantalite) concentrate, while lithium, primarily hosted within spodumene, was not recovered.

2.1 Corporate Structure

Tantalex was originally named Tantalex Resources Corporation, which was founded on 21 October 2013. Effective May 26, 2022, Tantalex Resources Corp. changed its name to Tantalex Lithium Resources Corp. to reflect the company's engagement in the acquisition, exploration, development and distribution of lithium, tantalum and other high-tech minerals.

On 23 March 2017, the Manono exploitation license (PER 13698) was awarded to MINOCOM, a joint venture between MINOR SARL and COMMINIERE SAS, which held 70% and 30% of MINOCOM respectively. Tantalex, via its 100% held Congolese subsidiary, Tantalex SAU, acquired 25% ownership of MINOCOM from MINOR on 2 July 2021, with an additional 27% acquired on 17 May 2022. TTX SAU currently holds Right of First Refusal on the remaining 18% of MINOR. The company structure for Tantalex is shown in Figure 2-1.

Figure 2-1
Tantalex Corporate Structure



2.2 Scope of Work

MSA was commissioned by Tantalex to complete a Mineral Resource Estimate on the Company's lithium-tin-tantalum tailings project (Manono Lithium Tailings Project) documented in an Independent Technical Report as per National Instrument 43-101 "Standards of Disclosure for Mineral Projects". Manono is located in Tanganyika Province of the Democratic Republic of Congo.

This Independent Technical Report has been prepared to comply with disclosure and reporting requirements set forth in the Toronto Venture Exchange (TSX-V) Corporate Finance Manual, Canadian National Instrument 43-101, Companion Policy 43-101CP, Form 43-101F1, the "Standards of Disclosure for Mineral Projects" (the Instrument) and the Mineral Resource and Reserve classifications adopted by CIM Council in May 2014.

2.3 Principal Sources of Information

MSA has based this Technical Report for the Manono Lithium Tailings Project on information provided by Tantalex along with other relevant published and unpublished data.

The Technical Report has been prepared on information available up to and including 10 July 2023, with the Mineral Resource having an effective date of 23 August 2023. The data used to estimate the Manono Tailings Mineral Resources represent the entire database for the drilling completed and there is no relevant material outstanding as of the effective date.

A personal inspection was made by the Qualified Person on 29 and 30 April 2022. The author has endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the Technical Report is based. A final draft of the Technical Report was also provided to Tantalex, along with a written request to identify any material errors or omissions prior to lodgement.

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

The locations of all maps are referenced to WGS 84, UTM Zone 35M, unless otherwise stated.

2.4 Qualifications, Experience and Independence

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

This report has been compiled by Rui Goncalves (BSc Hons, MSc (Eng.)), who is a geologist with 13 years' varied experience in the mining industry which includes exploration, mining geology and Mineral Resource estimation. He 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 (MGSSA). Rui Goncalves 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).

Neither MSA, nor the author of this report, has or has had previously any material interest in Tantalex or the mineral properties in which Tantalex has an interest. Our relationship with Tantalex 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.

3 RELIANCE ON OTHER EXPERTS

MSA has not independently verified, nor is it qualified to verify, the legal status of these concessions. The present status of tenements listed in this report is based on information and copies of documents provided by Tantalex, and the report has been prepared on the assumption that the tenements will prove lawfully accessible for evaluation. These documents include:

- 3.1 Tailings Licence PER 13698, Tantalex Resources
- Acte de Cession – TTX Minor – DRC-20210207
- PER13698 – 2022 Surface Rights – 50% payable to CAMI-MINOCOM-ND-DF-01784 DFA_2022
- PER13698 – 2022 Surface Rights – 50% payable to DGRAD-MINOCOM-NP-H3781185

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

Similarly, neither MSA nor the authors of this report are qualified to provide comment on environmental issues associated with the Tantalex Projects.

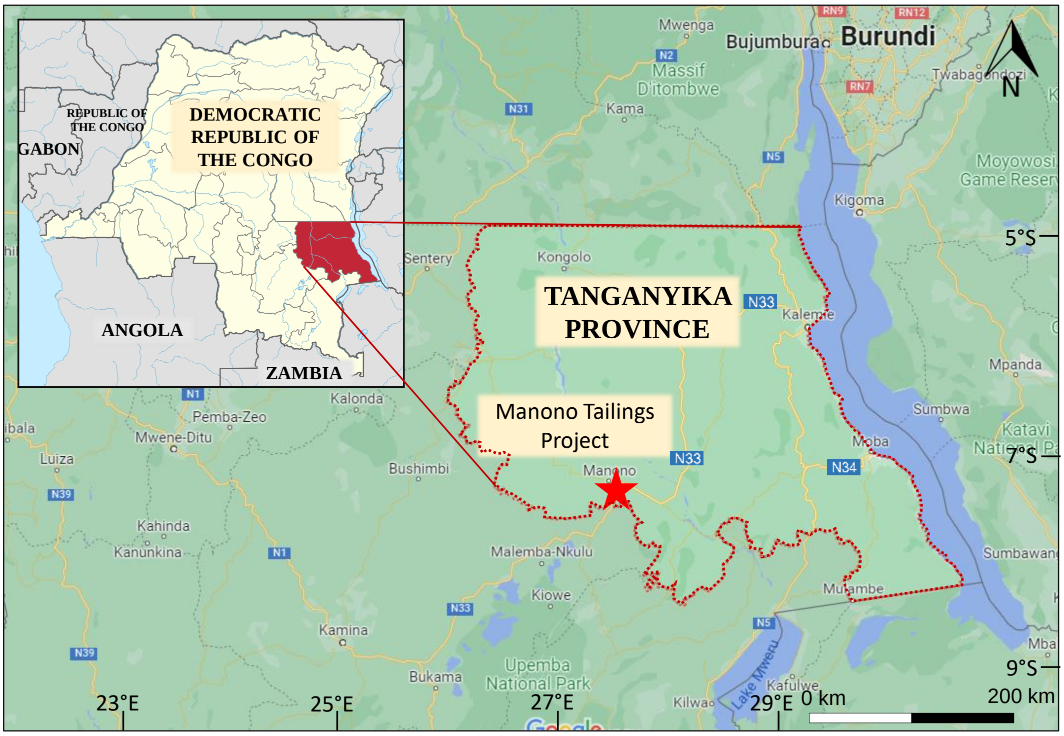
4 PROPERTY DESCRIPTION AND LOCATION

The Manono Lithium Tailings Project deposits are technogenic in nature, formed from the deposition of concentrator discard material created from processing of ore mined from the adjacent Manono tin mine. A total of approximately 100 million m^3 of material was mined from eluvial and weathered pegmatites between 1919 and 1982 (AVZ, 2017).

4.1 Location

The Manono Lithium Tailings Project is located directly south of the town of Manono, in the Tanganyika Province of the Democratic Republic of the Congo (DRC). The Project is located approximately 490 km north of the city of Lubumbashi, the second largest city in the DRC. The mining settlement towns of Manono and Kitotolo are partially located within the license boundary, to the west and east respectively. The Project is approximately located at a latitude of 7°17'S and a longitude of 27°24'E. The regional Project location is presented in Figure 4-1.

Figure 4-1
Regional Project location



***Source**: Adapted from Wikipedia and Google Maps (2022)*

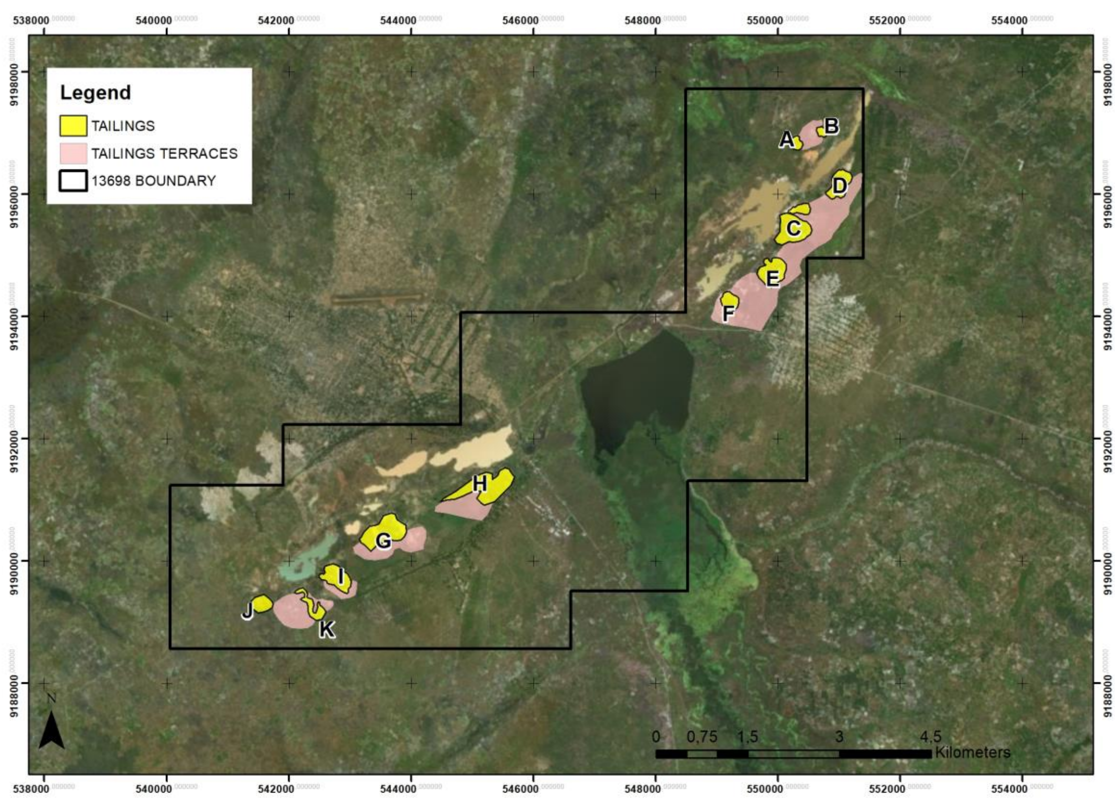
The Project consists of 11 coarse tailings dumps divided into a northern and southern sector and named alphabetically from A to K. The Northern Manono Sector contains dumps A to F while the Southern Kitotolo Sector contains dumps labelled G to K. A 12th overburden dump, labelled dump J, consists of laterite only. A fine tailings terrace is located directly adjacent to the coarse tailing dumps. The tailings dumps are labelled with a suffix "c" and the adjacent fine fraction is labelled "f".

Estimates were generated for 8 tailings dumps as listed below, of which five constitute Mineral Resources:

- C coarse (Cc)
- E coarse (Ec)
- H coarse (Hc)
- H fine (Hf)
- G coarse (Gc)
- G fine (Gf)
- I coarse (Ic)
- K coarse (Kc or just K)

The positions of the tailings deposits relative to one another are shown in Figure 4-2.

Figure 4-2
Manono Lithium Tailings Project area

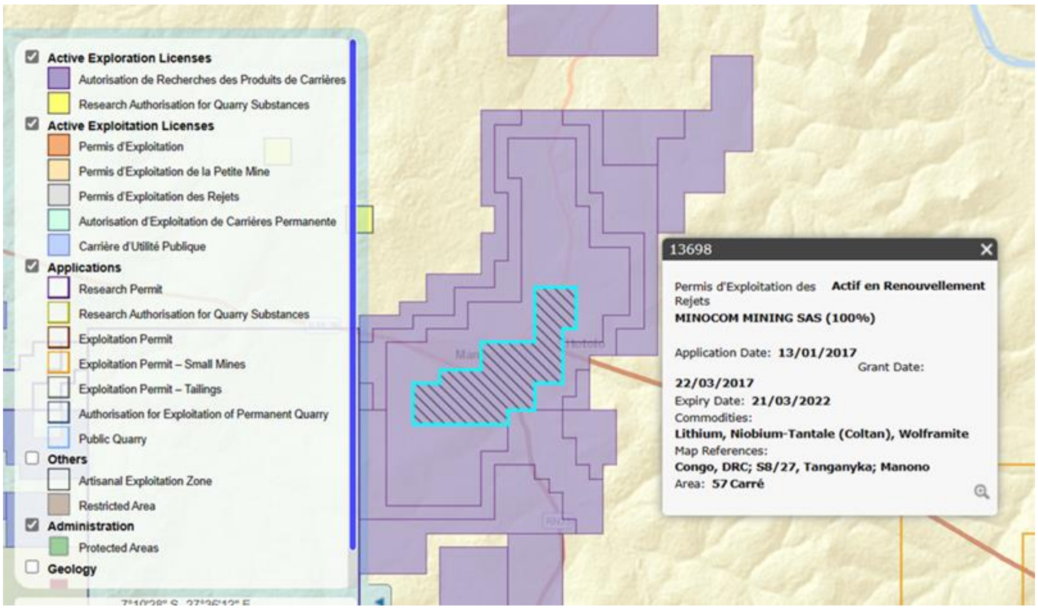


Source: *Tantalex (2022)*

4.2 Mineral Tenure, Permitting, Rights and Agreements

Tailings Exploitation Permit PER 13698 covers 57 km^2 (Figure 4-3) and is held by Minocom Mining SAS, a joint venture with 52% held by Tantalex, 18% held by MINOR and 30% held by the state-owned company COMINIERE. The permit was granted on 23 March 2017. Tailings exploitation licenses are renewable every 5 years and require the submission of an environmental and technical study. A renewal of the current license has received ''Avis favorable'' from the Mining Cadastre (CAMI) in July 2023 and currently awaiting the same from the Ministry of Mines. Once received, this will essentially grant the renewal of the licence for an additional period of 5 years. The Mining Code stipulates that upon renewal every 5 years, the Concession holder must give to the Government 5 % of the ownership in the concession.

Figure 4-3
Manono Lithium Tailings Project license area



Source: *http://drclicences.cami.cd/EN/ (2023)*

4.3 Surface rights

The DRC government has exclusive rights to all land but can grant surface rights to private or public parties. Surface rights are distinguished from mining rights and are payable in the event of granting a mining or quarry exploitation right as an annual fee per quadrangle. A mining right does not imply the right for any surface occupation over the surface, other than what is required for the operation.

The 2002 Mining Codes and its amendments, states that, subject to any rights of third parties over the surface concerned, the holder of an exploitation mining right has the right to occupy within the granted mining perimeter the land necessary for mining and associated industrial activities, including the construction of industrial plants and dwellings, water use, dig canals and channels and establish means of communication and transport of any type.

Occupation of land that deprives surface right holders of using the surface, or any modification rendering the land unfit for cultivation, entails an obligation on the part of the mining rights holder to pay fair compensation to the surface right holders. The mining rights holder is also liable for damage caused to the occupant's land due to any mining activity, even if such activity has been permitted and authorised.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

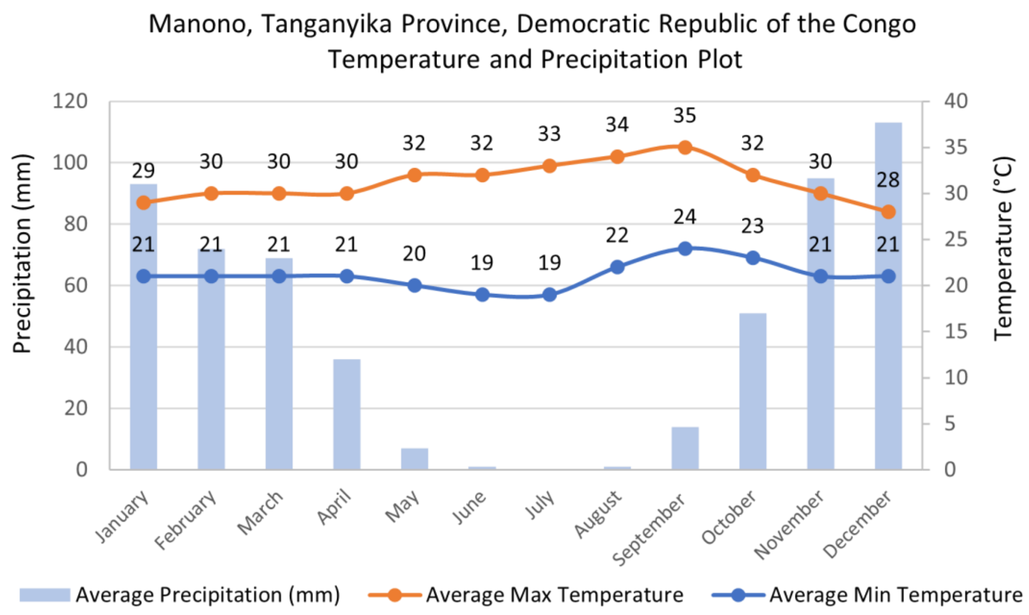
5.1 Topography, Elevation, Drainage and Vegetation

The topography of the Manono Lithium Tailings Project area is generally flat with an average elevation of 635 metres above mean sea level (mamsl). The tailings dumps reach a maximum height of approximately 70 m above the surrounding plains. The region supports a variety of vegetation that ranges from dense humid forest and clear forest to savannah and meadowlands. Within the Congo River Basin, the Lukushi river runs from south to north through the Tanganyika Province, passing the towns of Manono and Kitotolo, shortly before joining the major Luvua River.

5.2 Climate

The Project area has a tropical savanna climate with warm temperatures year-round (Figure 5-1). The wet season typically runs from October to March with an average of 19.2 rainy days per month and approximately 1 200 mm of rainfall per year. The dry season typically runs from April to September. The climate is not expected to affect the length of the operating season which typically runs throughout the entire year. Heavy rainfall may occasionally affect access to the site.

Figure 5-1
Manono temperature and precipitation plot



Source: *https://www.meteoblue.com/en/weather/historyclimate/climatemodelled/manono_dr-congo_209598 (2022)*

5.3 Access

Access to the Manono Lithium Tailings Project area is gained from Lubumbashi via a scheduled 1.5-hour flight to a small airport in Manono. Access may also be gained via road however wet weather conditions may affect road conditions. The road route from Lubumbashi to Manono is approximately 630 km.

The Project is approximately 215 km south of the Kongolo Railway station on the Great Lakes Line (Second Section). The national railway line is mostly operated by the Société Nationale des Chemins de Fer du Congo (SNCC). Railway lines are not all linked but are generally connected by river transport.

5.4 Local Resources and Infrastructure

Infrastructure in the adjacent mining towns of Manono and Kitotolo is currently limited. Power supply is generated by a solar power plant that was commissioned in March 2018. The solar power plant is the largest off-grid solar power plant in the region and supplies a new isolated network of the Société Nationale d'Electricité (SNEL). The production capacity is 1 MWp (megawatt peak). Since 2018, a hospital, a school, the airport, shops and housing are now connected to electricity (Groupe Forrest International, n.d.)

Water supply is in abundance for both local use and mining activities.

6 HISTORY

The Manono tailings originated from the processing of lithium-caesium-tantalum (LCT) enriched pegmatite material from the historical Manono-Kitolo mine, which operated from 1919 to the mid-1980's. In total, it is estimated that the mine produced 140,000 to 185,000 tonnes of tin and 4,500 tonnes of coltan concentrate, while lithium, in the form of spodumene, was not recovered (Scholtz, 2019).

6.1 Prior Ownership History of the Manono Lithium Tailings Project

La Congolaise d'Exploitation Minière S.A. (''Cominiere'') is a state-owned enterprise created April 12th, 2010 under the Ministry of Portfolio to manage and add value to the assets and concessions previously held by Zaire Etain. Zaire Etain was the last producer of the historical Manono-Kitotolo mine. PER13698 was initially held through a Cominiere JV held by Manomin as part of the PE12202. PE12202 expired on March 22, 2017 and was subsequently separated into two licenses by Cominiere, PER13698 which became the object of the JV MINOCOM MINING SAS and also PR13359 which was held until recently by DATHCOM MINING SAS.

Tantalex is unaware of any previous exploration work related to lithium undertaken on PER 13698 pertinent to the tailings.

6.2 Historical Mineral Resources and Reserves

Mineral Resources and Reserves have not been previously declared for the Manono Lithium Tailings Project.

6.3 Previous Production

There are no records of previous production from the tailings.

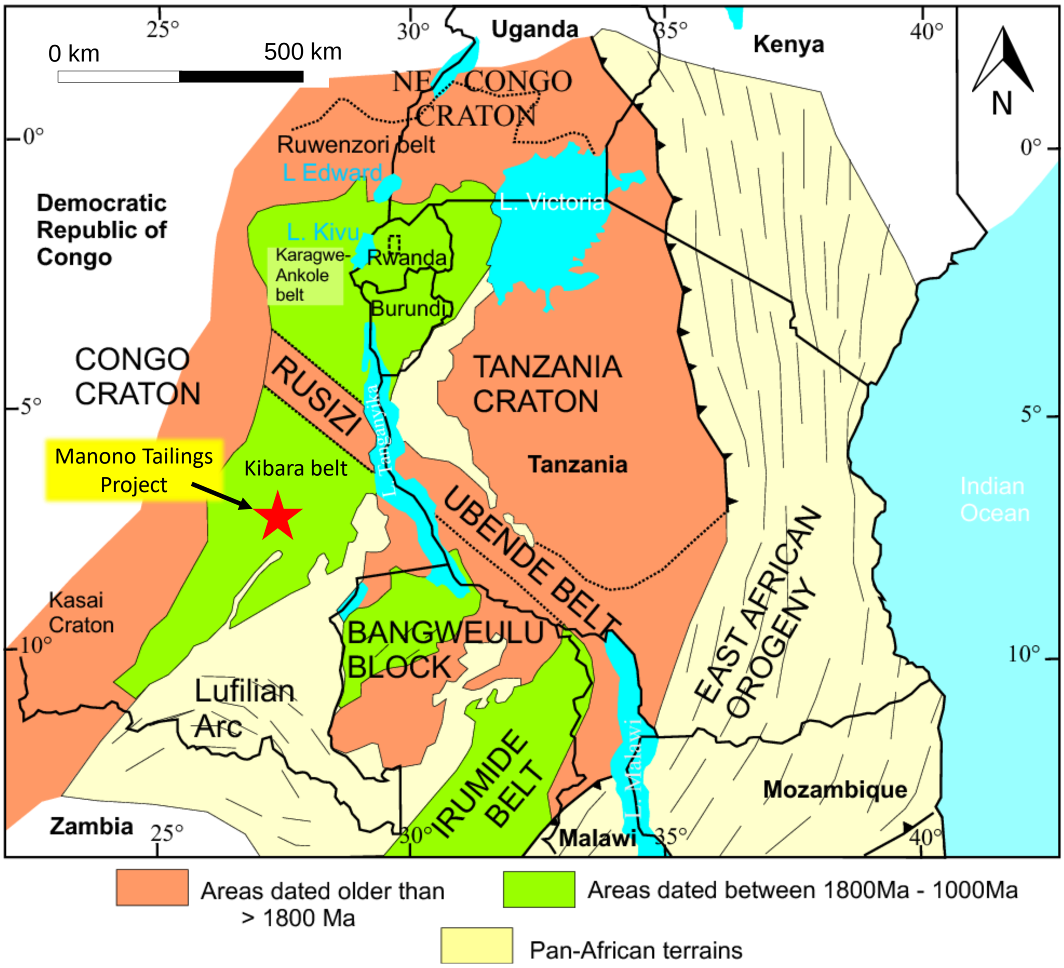
7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Manono Lithium Tailings Project is located within the northeast-southwest trending Central African Kibara Belt, which together with the Karagwe-Ankole Belt, a Mesoproterozoic intracratonic mobile belt, extend over 1 300 km from Katanga in the DRC to southwestern Uganda through Rwanda and Burundi (Figure 7-1). The southern Kibara and northern Karagwe-Ankole Belts formed between the Archaean-Palaeoproterozoic Congo craton to the west and north, the Archaean Tanzanian Craton to the east, and the Bangweulu Block to the south. Both the Kibara and the Karagwe-Ankole Belts form a large metallogenic province that hosts a variety of granite-related Sn-W-Nb-Ta mineralisation.

The Central African Kibara Belt comprises Palaeo- and Mesoproterozoic clastic sediments with minor metavolcanic rocks that have been intruded by multiple generations of granitoids ranging in age from approximately 1.4 Ga to 1.0 Ga. The oldest peraluminous granitoids (G1 and G2 granitic orthogneisses) were emplaced between 1.40 Ga and 1.38 Ga during an accretionary stage. The post-orogenic S-type tin-bearing granites (G4 Granites), and associated Sn-Ta-Nb-Li bearing pegmatites, veins and greisen bodies, intruded from 1.00 Ga to 0.95 Ga. The G4 Granites intruded the older Kibaran orthogneisses as well as the Kibaran metasedimentary units during continental collision and post-orogenic uplift (Pohl et al., 2013, Kokonyangi et al., 2006). A number of small stocks of this granite occur in the immediate vicinity of the workings at Manono and Kitotolo sectors (Dewaele et al., 2016).

Figure 7-1
Manono Tailing Project regional geology



***Source**: Adapted from Dewaele et al. (2013)*

Structural orientations are related to two major deformation events, D1 and D2. The D1 deformation resulted in an east-west to northeast trending fabric southwest of Manono, which changes to a northeast to north-northeast orientation in the Kalima area. The D2 deformation resulted in a northeast to north-northeast trending fabric. The mineralised veins and pegmatites are frequently orientated parallel to the northeast trending D2 fabrics, although some may have northwest, southeast or east-west orientations (Kokonyangi, 2004 and Kokonyangi et al., 2006).

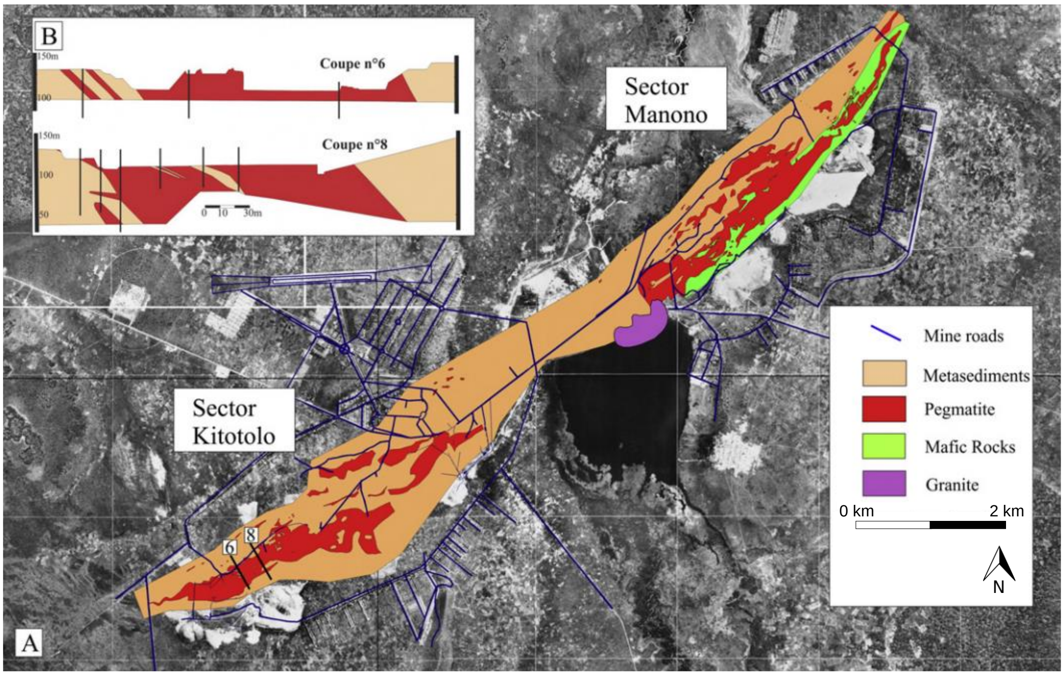
The Manono-Kitotolo deposit is considered the largest pegmatite hosted tin-columbite-tantalite-spodumene deposit in the DRC and one of the largest in the world (Dewaele et al., 2016). Dewaele et al., (2016) dated it at approximately 940 Ma which is consistent with the ages of the postulated parental G4 (tin) Granites and other pegmatites in the region.

Weathering and erosion of the quartz vein- and pegmatite-hosted tin and columbo-tantalite mineralisation has resulted in the significant alluvial and eluvial deposits in the recent and palaeo-drainage basins and floodplains throughout the region.

7.2 Local Geology

The Manono tailings are composed of concentrator reject material from processing of the Manono-Kitotolo deposit mined from various open pits that extend over the Manono-Kitotolo deposit area of approximately 800 m by 15 km (Figure 7-2). The Manono-Kitotolo deposit consists of two zones, the Manono-Kuhungwe Sector in the northeast and the Kitotolo Sector in the southwest, separated by the 2 km wide artificial Lake Lukushi.

Figure 7-2
Manono Lithium Tailings Project local geology



Source: *Adapted from Dewaele et al. (2016)*

Several large pegmatite intrusions have been recognised in the Manono-Kitotolo Sector along with numerous smaller pegmatite intrusions. The Roche Dure pegmatite is the largest intrusive body in the Kitotolo Sector with a strike length of at least 2 800 m and a width of 250 m. Pegmatites occur within the phyllitic or mica-schist host rocks with minor meta-sandstone horizons. In the Manono-Kahungwe Sector, pegmatites crosscut meta dolerites. The general strike of the pegmatites is at a bearing of 055° with a dip varying from 50°N to 50°S but predominantly subvertical (Dewaele, 2016). The pegmatite-metasediment contact shows minor small-scale folding but is largely parallel to the regional foliation orientation (Dewaele, 2016).

7.3 Project Geology

The Manono Lithium Tailings Project is composed of nine coarse tailings dumps and fine tailings terraces produced from mining and processing of material from the various Manono-Kitotolo open pits. The tailings material is typically coarse, ranging from 1 mm to 5 mm sized gravel as shown in Figure 7-3.

Figure 7-3
Manono Lithium Tailings Project coarse tailings material

Source: *Goncalves (2022)*

The material composition of each tailings deposit varies, with many being composed of a combination of pegmatite, laterite and/or clay material. Figure 7-4 illustrates the heterogeneity of the deposits, as observed for the Ic deposit. The contrast of the two material types is noticeable with the reddish-brown laterites juxtaposed against white pegmatite material. The J deposit is visible in background which consists exclusively of laterite material.

Figure 7-4
Ic deposit (looking southwest) illustrating the mixed nature of the materials making up these deposits

***Source**: Goncalves (2022)*

Few deposits appear to consist of a single material type, the exception to this being the K dump which is primarily composed of pegmatite. The K dump consists of tailings lying over a flat area 675 m by 500 m in extent, with depths up to 15 m in the centre, gradually thinning out to 3 m along the edges. Stacked tailings, up to 20 m high are located in the northwest corner of the K dump, while stacked tailings in a cone-like shaped feature are found in the east of the deposit, attaining a maximum thickness of 45 m. Figure 7-5 shows the white, pegmatite tailings and the partially vegetated cone-like feature of the K dump.

Figure 7-5
K dump with the stacked, cone-like feauture of the K dump (looking south)

***Source**: Goncalves (2022)*

Fine vegetation, consisting of shrubs and tall grass, covers the majority of the tailings deposits. This tends to be thicker in the lower lying tailings of the K, Gf and Hf deposits. Some deposits show evidence of historical and recent artisanal mining activity for cassiterite and coltan as observed by the disturbed ground in the foreground of Figure 7-6 (K deposit).

Figure 7-6
Pegmatite tailings of the K dump, illustrating vegetation cover and historical artisanal mining in the foreground

***Source**: Goncalves (2022)*

7.4 Mineralisation

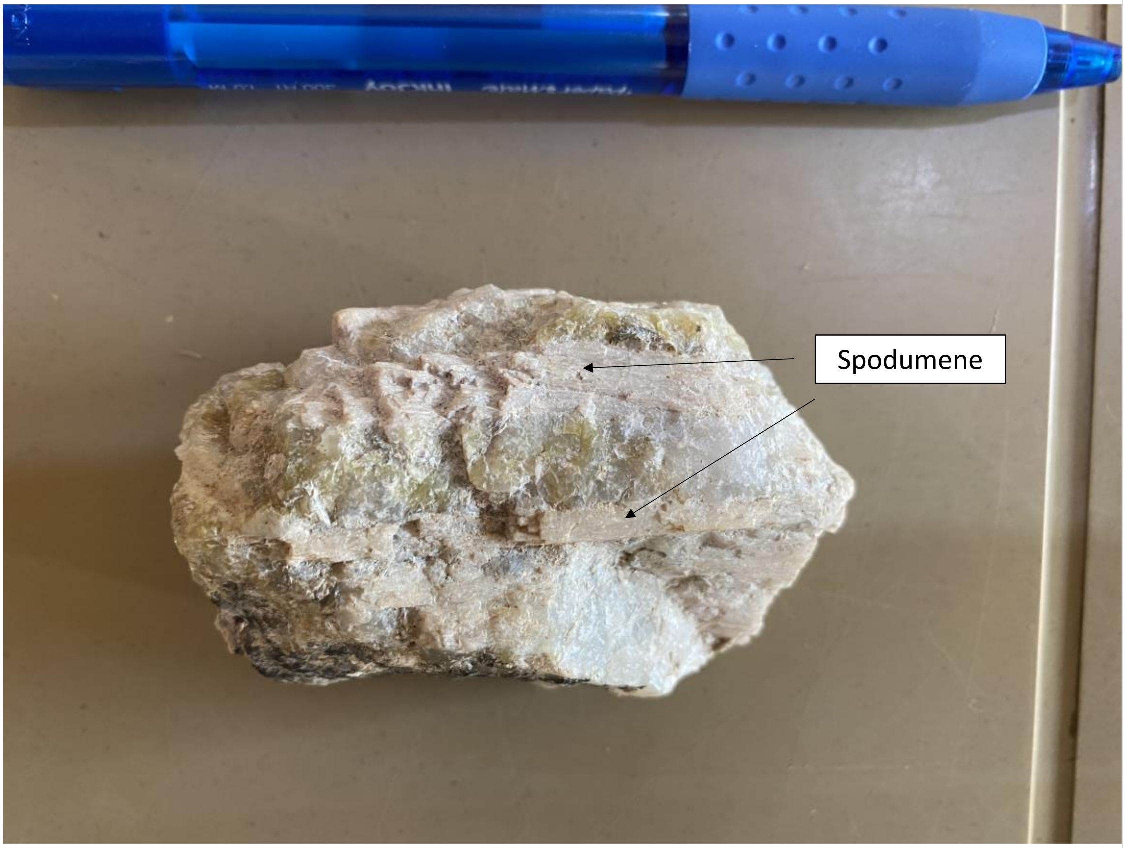
The Manono-Kitotolo mine exploited a large pegmatite deposit that produced between 140 000 tonnes and 185 000 tonnes of tin and 4 500 tonnes of coltan concentrate (Scholtz, 2019). The reject processed material was deposited on the coarse tailings dumps and fine tailings terraces that make up the Manono Lithium Tailings Project.

Lithium is present in the minerals spodumene and lepidolite, and tin is present in cassiterite. The tailings still contain cassiterite currently being mined by artisanal miners. The majority of the pegmatites mined also contain spodumene (and/or lepidolite) and the minerals can be visually identified in the material on the coarse tailings dumps (Scholtz, 2019). The relatively high grade of lithium in spodumene was analysed in two grab samples by BRGM (1.7% to 2% LiO_2) and indicates that lithium was likely not recovered during historical processing (Scholtz, 2019).

A centimetre sized sample of pegmatite recovered from the project area is illustrated in Figure 7-7. This shows visible spodumene crystals which can be easily identified by the presence of prismatic cleavage.

Figure 7-7

Pegmatite sample from Manono illustrating prismatic cleavage of spodumene



Source: *Goncalves (2022)*

8 DEPOSIT TYPE

The Manono Lithium Tailings Project is composed of the reject LCT (Lithium-Caesium-Tantalum) pegmatite material processed at the Manono-Kitotolo mine from 1919 to the mid-1980s. Technogenic deposits are a category of superficial formations created by anthropogenic direct or induced depositional processes.

Tailings from the Manono-Kitotolo open pits were deposited on the ground adjacent to the various open pits. The coarse tailings were deposited over many years into raised heaps that reach heights of up to 70 m above surface. The fine tailings material was deposited into flat terraces adjacent to the coarse tailings dumps.

Many of the tailings deposits are composite in nature, consisting of layers of pegmatite, laterite and/or clay layers. These were deposited by mechanical means, including most of the deposits denoted as “fines”, with the exception of the Hf and Gf deposits, which are assumed to have formed due to the settling of fine material in standing ponds of water as evidenced by the presence of clay layers in these deposits.

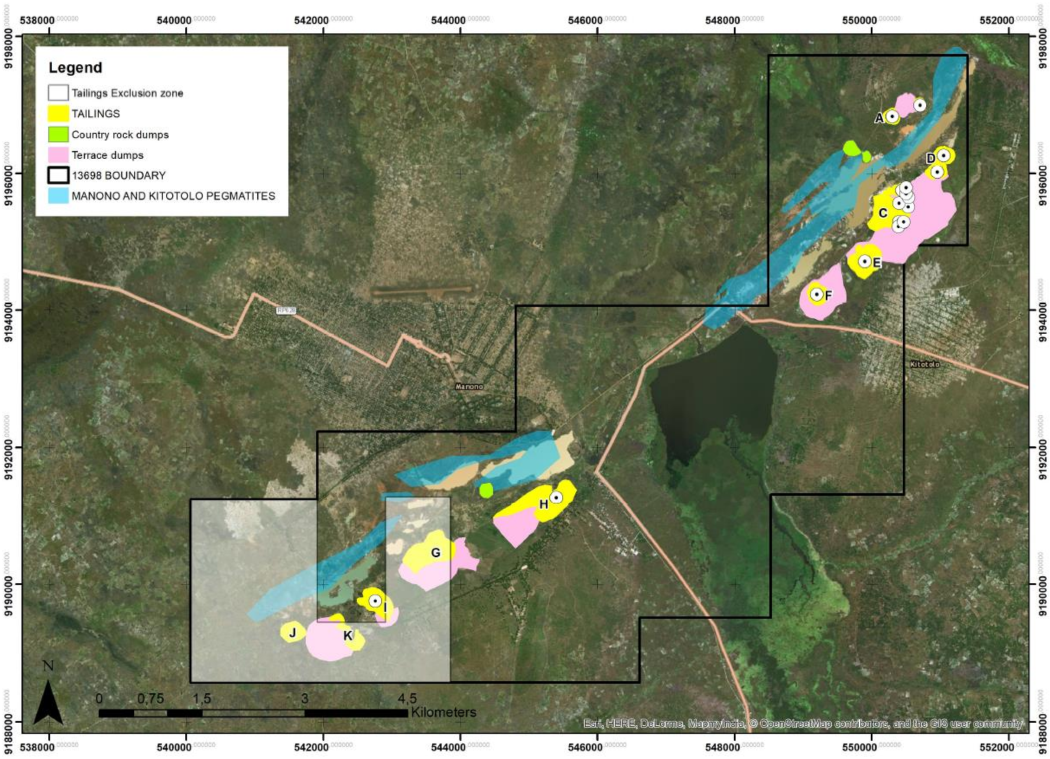
Technogenic deposits such as those at Manono are typical of many mining operations across the globe and often contain concentrations of various metals of economic value due to incomplete recovery during the processing of the raw, in-situ source material. Their extents and depths tend to be well defined and due to their recent formation, the only processes affecting their evolution is erosion due to fluvial or aeolian processes.

9 EXPLORATION

9.1 Previous Exploration

In 2019, a grab sampling program was conducted by Nico Scholtz (a consultant to Tantalex) and the Tantalex field team. The grab sampling was conducted on ten of the coarse tailings dumps and associated fine tailings terraces as indicated in Figure 9-1.

Figure 9-1
Location of grab samples



Source: *Scholtz (2019)*

In total, 43 grab samples were taken from various parts of all the tailings and tailings types. These included tailings dump gravel, lepidolite, various spodumene samples, spodumene pegmatite and weathered samples. Grab samples are not considered representative of the Manono Lithium Tailings Project's mineralisation and do not form part of the Mineral Resource estimate. The purpose of the grab sampling was solely for identifying the presence of mineralisation and the more prospective dumps.

9.2 Bulk Sampling

An initial bulk sampling program was conducted by Nico Scholtz and the Tantalex field team. The bulk sample was collected from the "C" dump and included both the coarse and fine material. The "C" dump was considered to be most representative of the tailings mineralisation and was most accessible at the time of sampling. Eighteen bulk sample bags with a weight of approximately 50 kg each were collected for metallurgical testwork purposes (Scholtz, 2019).

9.3 Cobra Drilling

Prior to the commencement of the Mineral Resource drilling campaign, Tantalex undertook a trial drilling campaign using a handheld Atlas Copco Cobra Combi rock drill, which was modified to hold a core barrel for sample collection. A total of 132 Cobra holes were drilled on four deposits, namely the C (56 drillholes), the G (16 drillholes), the H (38 drillholes) and the K (22 drillholes) dumps, totalling 967.8 metres of drilling.

Twenty-two of the Cobra drillholes, representing 101.3 m of drilling, were sampled and assayed. Samples were taken at 3 m intervals, which resulted in 19 samples from 8 drillholes that were submitted to ALS, and 19 samples from 14 drillholes that were submitted to SGS Johannesburg. The samples were all taken from the K dump, results for which are presented in Table 9-1.

Table 9-1
Results of Cobra drilling programme

Drillhole ID	Depth from m	Depth to m	Li_2O %	Sn ppm	Ta ppm
MDC046	0	2.7	0.88	87	10.3
MDC049	0	3	1.26	432	35.8
MDC049	3	4.5	1.16	309	30.2
MDC051	0	3	0.01	79	8.0
MDC051	3	6	0.01	185	13.1
MDC052	0	2.6	0.85	198	22.9
MDC052	2.6	4.3	0.76	258	30.5
MDC053	0	2.5	1.71	443	33.7
MDC055	0	3	1.37	576	46.8
MDC057	0	3	1.27	558	43.6
MDC058	0	3	1.27	454	36.7
MDC059	0	3	0.63	253	23.0
MDC060	0	3	1.45	439	34.0
MDC062	0	3	1.53	394	36.0
MDC062	3	6	1.09	258	35.4
MDC063	0	3	1.49	574	47.6
MDC065	0	3	1.56	508	49.3
MDC065	3	6	1.36	350	31.7
MDC067	0	2	0.34	217	21.1
MDC047	0	3	1.13	320	25.1
MDC047	3	5.6	0.98	382	22.5
MDC048	0	3	1.10	382	24.8
MDC048	3	6	0.14	243	15.9
MDC048	6	7	0.02	176	5.8
MDC050	0	3	1.12	457	24.7
MDC050	3	6	1.01	388	22.5
MDC054	0	3	0.75	779	38.0
MDC054	3	6	0.93	527	37.3
MDC056	0	3	1.09	626	49.4

Drillhole ID	Depth from m	Depth to m	Li_2O %	Sn ppm	Ta ppm
MDC056	3	6	1.13	726	35.1
MDC056	6	7	1.44	673	36.5
MDC061	0	3	1.13	499	29.2
MDC061	3	5.8	1.21	395	26.8
MDC061	5.8	6.7	1.09	320	25.5
MDC064	0	3	1.06	557	27.7
MDC064	3	5	1.24	522	35.1
MDC066	0	3	1.07	336	22.2
MDC066	3	6	0.56	352	23.3

The results of the Cobra drilling were not used for Mineral Resource estimation due to the limited penetration into the dump. However, they provided an indication of the magnitude of the grade of tin, tantalum and lithium mineralisation in the four dumps and the motivation to carry out a Mineral Resource drilling programme.

9.4 Geophysical Survey

In 2017, an aeromagnetic geophysical survey was conducted over the Manono Lithium Tailings Project area by International Geoscience Services (IGS), funded by the World Bank PROMINES project in support of the Ministry of Mines of the DRC. The high resolution regional airborne survey was flown by New Resolution Geophysics (NRG) from South Africa, at a line spacing of 400 m, with selected targets being surveyed at a closer spacing of 200 m. IGS was responsible for the management and technical coordination of the project on behalf of the Ministry. Tantalex does not have access to the report resulting from this survey but has acquired the associated data, which is of no direct relevance to the tailings deposits.

10 DRILLING

Drilling at the Manono Lithium Tailings Project began in September 2021 and was completed In July 2022 using a track mounted aircore/RC rig with an onboard compressor. Aircore drilling was undertaken using an 80 mm outer diameter core bit and a 30 mm inner core diameter bit. A geologist was present throughout the drilling operation to supervise both the drilling and sampling process.

Drilling took place in two phases, with the first phase ending in November 2021. Tantalex subsequently decided to undertake further drilling, with the intention of providing closer spaced drilling information for higher confidence estimates for several deposits. Phase 2 commenced on 15 June 2022 and concluded on 8 July 2022.

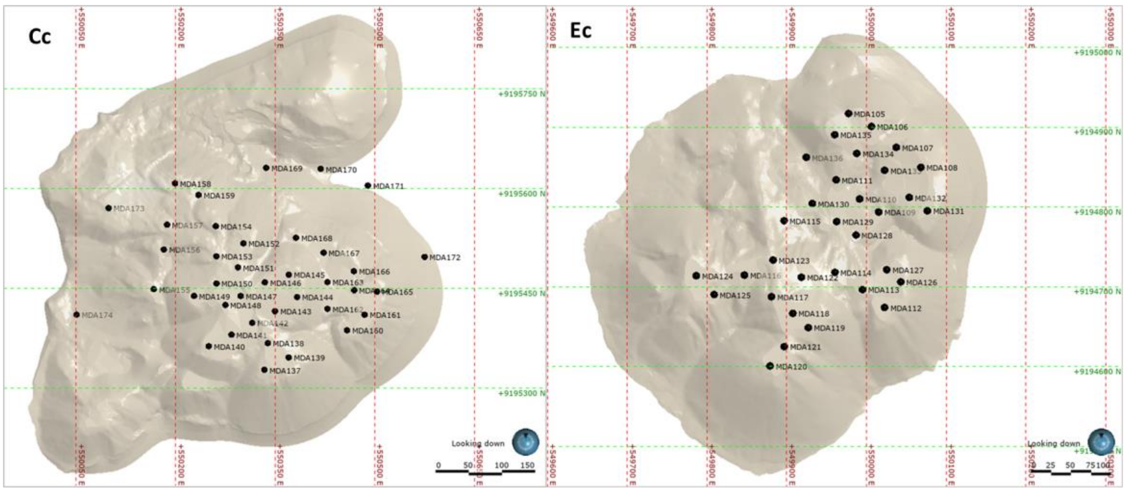
A summary of the two phases of the Tantalex drilling campaign is presented Table 10-1.

Table 10-1
Tantalex drilling campaign summary

Phase	From	To	Type	Number of Drillholes	Metres Drilled
Phase 1	September 2021	November 2021	Aircore	174	9 279.9
Phase 2	June 2022	July 2022	Aircore	194	2 657.0

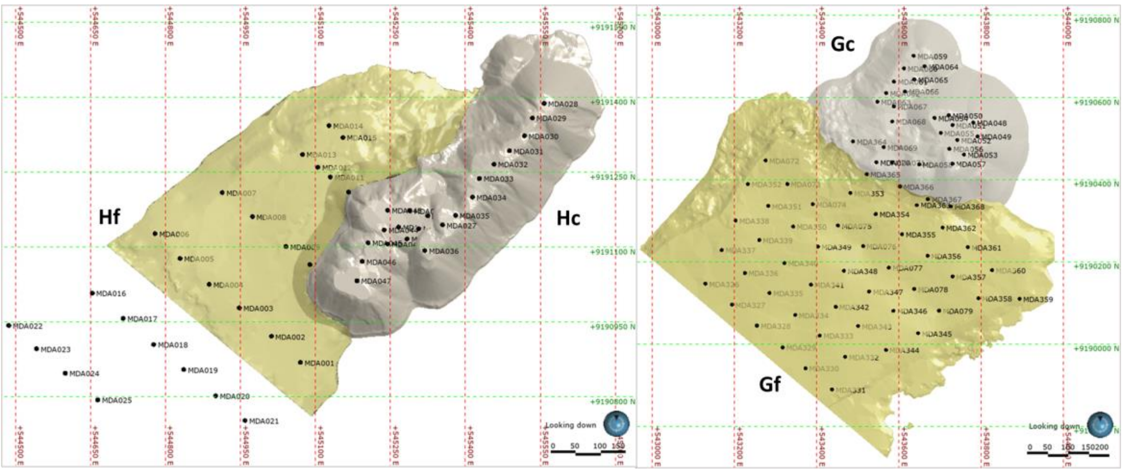
The collar locations for the drilling campaign are presented in Figure 10-1 for the Cc and Ec deposits, Figure 10-2 for the Hc, Hf, Gc and Gf deposits, and Figure 10-3 for the Ic and K deposits.

Figure 10-1
Manono Lithium Tailings Project drillhole collars for the Cc and Ec deposits



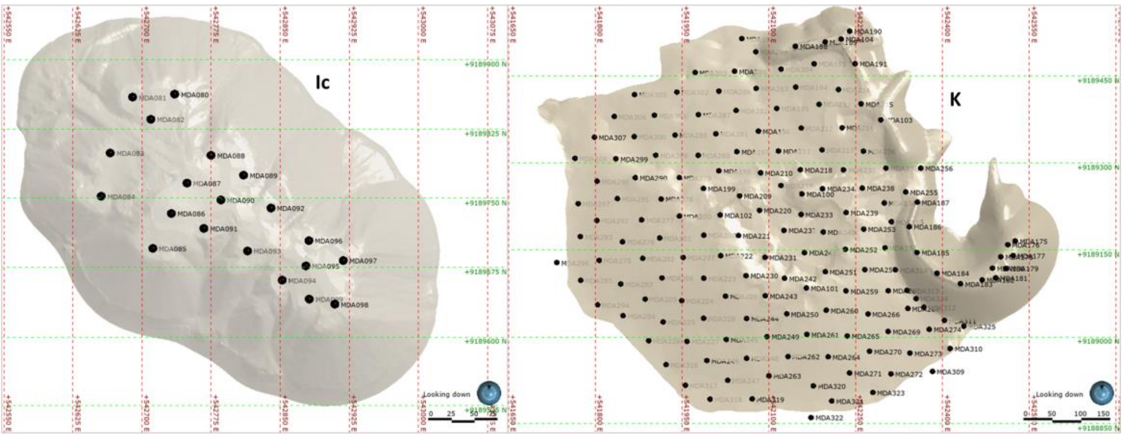
Source: *MSA (2022)*

Figure 10-2
Manono Lithium Tailings Project drillhole collars for the Hc, Hf, Gc and Gf deposits



Source: *MSA (2022)*

Figure 10-3
Manono Lithium Tailings Project drillhole collars for the Ic and K deposits



Source: *MSA (2022)*

10.1 Drillhole Sample Recovery

The weight of each aircore sample was recorded and used as a proxy to calculate an average sample recovery. On average, each sample weighed between 2.5 kg to 5 kg, with an average recovered weight of 3.9 kg.

10.2 Collar Surveys

The collar coordinates were surveyed on completion of the hole using a Trimble R4s GNSS (Global Navigation Satellite System) and were captured in the WGS84 UTM35S Zone geodetic system. The Trimble R4s utilises signals from all six GNSS and produces a Real-time Kinematic position (RTK) with a horizontal accuracy of 8 mm and a vertical accuracy of 15 mm (Optron, 2022).

The drillhole collars were marked with a concrete beacon recording relevant details of each hole as shown in Figure 10-4.

Figure 10-4
Concrete plinth over collar MDA050

Source: *Goncalves (2022)*

10.3 Downhole Surveys

All holes were drilled vertically with an approximate average depth of 32 m and a maximum depth of 86 m. Downhole surveying to check hole deviation was deemed not necessary as minimal deviation is expected to occur.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Logging

The geologist logged a wet sieved (+3 mm) portion of between 200 g to 300 g of each 1 m sample interval (Figure 11-1). The sieved sample was transferred to a plastic chip tray prior to detailed logging directly into a Microsoft Excel spreadsheet. Each chip tray contains coarse- and fine-grained material as a representation of the 1 m sampled interval (Figure 11-2).

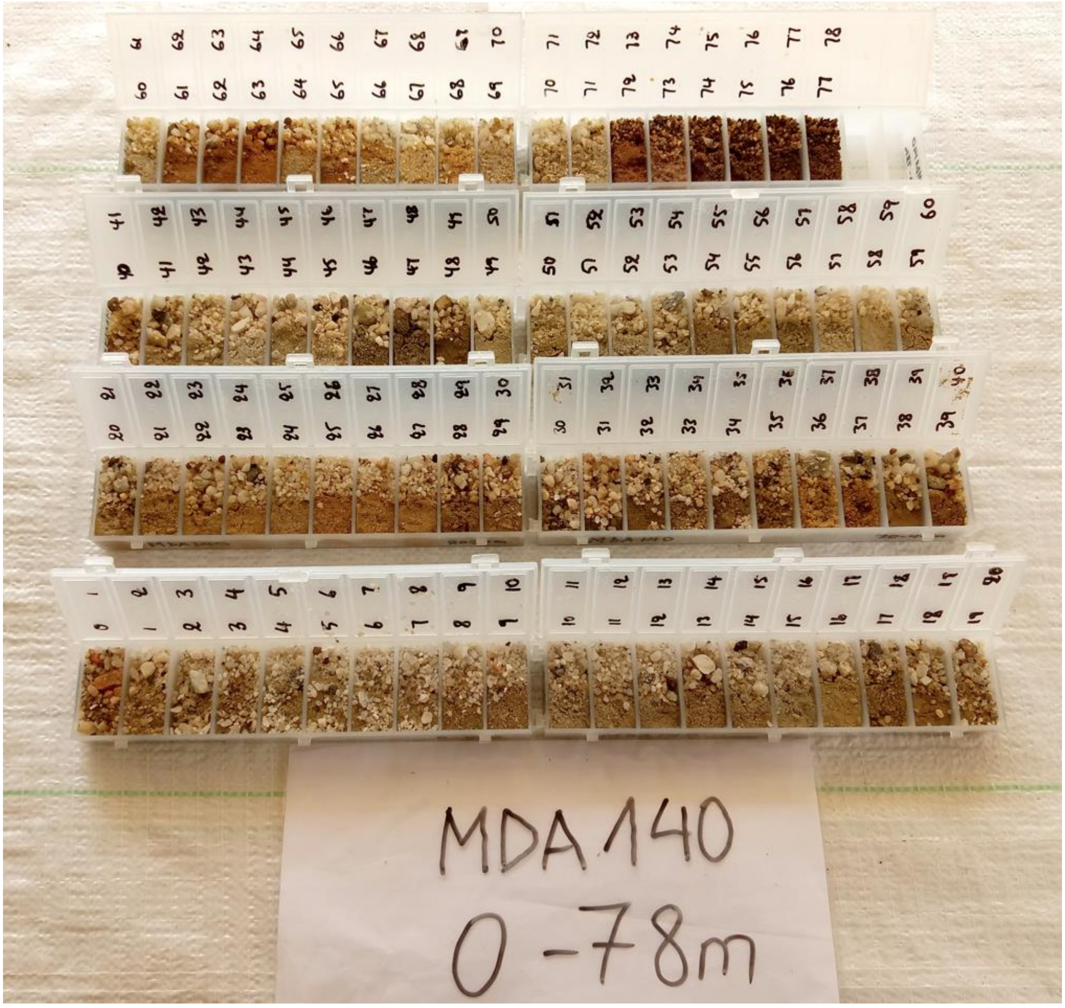
Figure 11-1
Manono Lithium Tailings Project geological logging

Source: *Lindhorst (2022)*

The geologist recorded the sample weight, lithology and colour. Any additional grain size, mineralisation and alteration information was generally recorded as a comment.

Once logged, the chip trays were photographed using a digital camera and images are stored on the Tantalex Dropbox™ file hosting service.

Figure 11-2
Manono Lithium Tailings Project chip tray photograph example



Source: *Tantalex (2022)*

11.2 Sample Handling

Samples weighing between 2.5 kg and 5 kg were collected at 1 m intervals in large polyweave bags from the rig-mounted cyclone (Figure 11-3).

Figure 11-3
Track mounted aircore rig with mounted cyclone

Samples were transferred into calico bags which were pre-labelled with the drillhole number and the relevant metre interval. Samples were laid out at the drill site in sequential order to ensure all 1 m sample intervals are accounted for and to check that all samples are correctly and clearly labelled. A list of the required QAQC samples to be inserted at regular, predetermined intervals was recorded by the geologist (Lindhorst, 2022, personal communication).

Samples were collected into larger 50 kg polyweave bags for transport by the Tantalex drivers to the Manono base camp for temporary storage before transportation to the on-site sample preparation facility. The Sample Preparation Facility Manager was responsible for organising the transport of the samples from the Manono base camp to the preparation facility (Lindhorst, 2022).

11.3 Sample Compositing

The 1 m sample intervals were prepared into 3 m composite samples. Each sample was passed through a Jones Riffle Splitter to halve the initial 1 m samples. The three half-samples were combined and mixed into a single composite sample and then riffle split down to 400 grams for pulverisation and assay. The reject half sample of each initial 1 m sample was returned to the original bag and retained for future reference (Lindhorst, 2022).

In the early stages of the drilling campaign, sampling was carried out at 1 m intervals, however, soon afterwards this was changed to 3 m composite samples as described above. From a total of

3271 samples, 1126 were taken at 1 m intervals, 90 samples were taken as 2 m composites and 2 samples were taken as 4 m composites (in both cases at the end of a drillhole), while the remainder (2053) of the samples were taken at 3 m intervals.

11.4 Sample Preparation

A geologist was responsible for ensuring that the Sample Preparation Facility Manager received the QAQC sample list for insertion of the required QAQC samples. The Tantalex Preparation Facility utilised one of three different protocols for sample preparation during the 2021-2022 drilling program. Sampling Protocol One (the original protocol) was utilised until the breakdown of the on-site sample pulveriser, after which Sampling Protocol Two was implemented. Sampling Protocol Three was further implemented after the breakdown of the on-site roll crusher (Lindhorst, 2022).

11.4.1 Sample Preparation Protocol One

- Samples were weighed and the weights recorded on paper for later digitisation into the 'Sample Preparation Data' Microsoft Excel spreadsheet by the Sample Preparation Facility Manager.
- Sample material was transferred from the calico sample bags to 40 cm by 60 cm sample drying trays. The sample ID was recorded onto a cardboard tag which was placed into the drying tray. The trays were placed onto a metal plate oven and heated for approximately 10 to 20 minutes by a wood burning fire (Figure 11-4).

Figure 11-4
Wood fire ovens and drying pans used to dry samples

- Once dry, samples were allowed to cool for approximately 10 minutes before being transferred back to the original calico sample bag, together with the sample tag.
- The dry samples were weighed and the weights recorded on paper for later digitisation into the 'Sample Preparation Data' Microsoft Excel spreadsheet.
- Samples were screened using a 5 mm sieve. The +5 mm size fraction was weighed and weights recorded for metallurgical purposes, after which it was added back to the sample.
- The entire sample was passed through a roll crusher to reduce the size to 2 mm.
- The crushed sample was passed through a Jones Riffle Splitter in order to obtain a 200 g sample.
- The 200 g sample was sub-sampled into a 100 g sample using the cone and quartering technique. The 200 g sample was homogenised by transfer between containers for three passes. The sample was then formed into a cone and flattened. The two 50 g opposite quarters were selected to make up the 100 g sample.
- The 100 g sample was pulverised to more than 80% finer than 75 µm.
- The pulverised samples were packaged into boxes with the inserted QAQC samples for transport to Lubumbashi.
- The 100 g reject sample was retained for future reference.
- A sample submission form was created in Lubumbashi for inclusion with the samples that were sent to ALS, Ireland by via FEDEX courier service.
- Sample dispatch details were entered into the Assay Register spreadsheet.

11.4.2 Sample Preparation Protocol Two

After the breakdown of the on-site pulveriser, samples were initially prepared as per Protocol One and crushed to reduce the sample size to 2 mm, thereafter:

- The entire 200 g sample was transferred into pulp paper sampling packet for transport to the COPROCO warehouse in Lubumbashi.
- A sample submission form for the ALS affiliated Congolese Analytical Laboratory SARL (COAL) Laboratory was created in Lubumbashi after sample checks.
- Samples were transported to the COAL Laboratory located at the SOMIKA mining site.
- The entire sample was pulverised using a LM3 ring mill to more than 85% finer than 75 µm.
- A 100 g sub-sample was transferred to a labelled, pulp paper sampling packet.
- The 100 g reject sample was placed into a labelled, zip-lock plastic bag.
- The 100 g pulp samples were packed by Tantalex into labelled ALS sample boxes for transport by FEDEX courier services to either ALS, Ireland or to SGS, Randfontein, South Africa. The reject samples are stored in boxes at the COPROCO locked warehouse facility.
- At ALS, Ireland, samples were re-pulverised to more than 85% finer than 75 µm (technique code PUL-31) to ensure homogenisation after transport.

- At SGS, South Africa, samples were re-pulverised to more than 85% finer than 75 µm to ensure homogenisation after transport.
- Reject pulps are stored in a locked room at the COPROCO Mineral Processing Warehouse located at 21 Nyanza Lubumbashi.
- Sample dispatch details were entered into the Assay Register spreadsheet.

11.4.3 Sample Preparation Protocol Three

- After the breakdown of the on-site crusher, samples were weighed, dried and screened as per Protocol One, thereafter the entire 400 g screened was transported to the COAL Laboratory Lubumbashi;
- The entire sample was crushed at the laboratory to a 2 mm size fraction using a benchtop jaw crusher.
- Reject preparation samples are stored at the COAL Laboratory for future retrieval.

Samples were couriered to either ALS, Ireland or SGS, Randfontein, as per Protocol One and Two.

11.5 Sample Analyses

The sub-samples were analysed at ALS, Ireland, (Irish National Accreditation Board (INAB) accreditation number 173T, ISO 17025) or SGS, South Africa (SANAS accreditation number T0265, ISO 17025). In total, 8 038 metres of core were sent for analysis, of which 28% were analysed at ALS. The primary laboratory was changed to SGS due to cost reasons.

At ALS, Ireland, samples were analysed using the following techniques:

- Super Trace Na_2O_2 by ICP-MS (technique code ME-MS89L), for Ag ppm, As ppm, Ba ppm, Be ppm, Bi ppm, Ca%, Cd ppm, Ce ppm, Ce ppm, Co ppm, Cs ppm, Cu ppm, Dy ppm, Er ppm, Eu ppm, Fe%, Ga ppm, Gd ppm, Ge ppm, Ho ppm, In ppm, K%, La ppm, Li ppm, Lu ppm, Mg%, Mn ppm, Mo ppm, Nb ppm, Nd ppm, Ni ppm, Pb ppm, Pr ppm, Rb ppm, Re ppm, Sb ppm, Se ppm, Sn ppm, Sr ppm, Ta ppm, Tb ppm, Te ppm, Th ppm, Ti%, Tl ppm, Tm ppm U ppm, V ppm, W ppm, Y ppm, Yb ppm and Zn ppm;
- Na_2O_2 fusion and ICP-AES for high-grades (technique code ME-ICP82b) for Li%;
- Lithium Borate Fusion and ICP-MS (technique code ME-MS81) for Ba ppm, Ce ppm, Cr ppm, Cs ppm, Dy ppm, Er ppm, Eu ppm, Ga ppm, Gd ppm, Hf ppm, Ho ppm, La ppm, Lu ppm, Nb ppm, Nd ppm, Pr ppm, Rb ppm, Sm ppm, Sn ppm, Sr ppm, Ta ppm, Tb ppm, Th ppm, Tm ppm, U ppm, V ppm, V ppm, W ppm, Y ppm Yb ppm, Zr ppm;

At SGS, South Africa, samples were initially analysed using the following technique:

- Na_2O_2 Fusion with HNO_3 acid digest, combined ICP-OES and ICP-MS (technique code GE_IMS90A50) for Ag ppm, Al%, As ppm, Ba ppm, Be ppm, Bi ppm, Ca%, Cd ppm, Ce ppm, Co ppm, Cr ppm, Cs ppm, Cu ppm, Dy ppm, Er ppm, Eu ppm, Fe%, Ga ppm, Gd ppm, Ge ppm, Ho ppm, In ppm, K%, La ppm, Li ppm, Lu ppm, Mg%, Mn ppm, Mo ppm, Nb ppm, Nd ppm, Ni ppm, P%, Pb ppm, Pr ppm, Rb ppm, S%, Sb ppm, Si%, Sm ppm, Sn ppm, Sr ppm, Ta ppm, Tb ppm, Te ppm, Th ppm, Ti%, Tl ppm, Tm ppm, U ppm, V ppm, W ppm, Y ppm, Yb ppm and Zn ppm.

Prior to the release of the maiden Mineral Resource estimate, MSA identified issues with the accuracy and precision of the tin and tantalum assays which resulted in a comprehensive internal review by SGS. Subsequently, the samples were re-submitted for repeat analyses for tin and tantalum, with Tantalex opting to re-assay samples from the K, Ic, Gc and Gf dumps only as the other deposits do not form part of the Mineral Resources. SGS concluded that the inconsistent results in the tin and tantalum assays were caused by incomplete furnace fusion at 600°C and poor stability using nitric acid as a leaching media. As a result, the analytical method for these two elements was adjusted, with a flame fusion and hydrochloric acid digest being used instead. Furthermore, an additional 66 previously un-assayed samples, representing 7 drillholes from the Ic dump, were included by Tantalex for lithium analysis.

11.6 Sampling Governance, Storage and Security

Geological samples are stored in the ten-sample plastic chip trays in sequential order in the sample warehouse on-site (Figure 11-5).

Figure 11-5
Manono Lithium Tailings Project chip sample storage

Source: *Lindhorst (2022)*

The reject half sample of each original 1 m sample interval was returned to the original bag and retained for future reference at the on-site preparation facility. All rejects from the 3 m composite samples are also stored at the on-site preparation facility (Figure 11-6).

Figure 11-6
Sample storage facilities and polyeave bags containing samples

The -2 mm crushed rejects prepared on-site are stored at the on-site preparation facility. All 100 g sample and pulp rejects from the Lubumbashi COAL Laboratory are stored at the COPROCO mineral processing facility in a locked storage room at 2 Nyanza Ave, Kampemba, Lubumbashi DRC. The sample rejects at the Manono site and in Lubumbashi will be kept indefinitely.

Sample rejects processed at ALS, Ireland, have been disposed of. Sample rejects processed at SGS, South Africa, are currently still available at the laboratory and will be disposed of on completion of the project.

11.7 Quality Assurance and Quality Control

Appropriate quality assurance and quality control (QAQC) monitoring is a critical aspect of the sampling and assaying process in any exploration programme. Monitoring the quality of laboratory analyses is fundamental to ensuring the highest degree of confidence in the analytical data and providing the necessary confidence to make informed decisions when interpreting all the available information. Quality assurance may be defined as information collected to demonstrate that the data used further in the Project are valid. Quality control (QC) comprises procedures designed to maintain a desired level of quality in the assay database. Effectively applied, QC leads to identification and corrections of errors or changes in procedures that improve overall data quality. Appropriate documentation of QC measures and regular scrutiny of quality control data are important as a safeguard and form the basis for the quality assurance programme implemented during exploration.

In order to ensure quality standards are met and maintained, planning and implementation of a range of external quality control measures is required. Such measures are essential for minimizing uncertainty and improving the integrity of the assay database and are aimed to provide:

- An integrity check on the reliability of the data.
- Quantification of accuracy and precision.
- Confidence in the sample and assay database; and
- The necessary documentation to support database validation.

The Manono QAQC programme reserved three in every twenty samples as QC samples (resulting in approximately 16% QAQC samples), usually one duplicate, one Certified Reference Material (CRM) and one certified blank sample.

11.7.1 Blank Samples

Certified blank sample material was purchased from African Mineral Standards (AMIS0439), consisting of silica chips. Blank samples were inserted at a frequency rate of approximately one in every twenty samples, although a lower, irregular frequency was used in the early stages of the exploration programme. The blank samples were subjected to the same sample preparation and analytical processes and were within the same sample stream as the routine field samples.

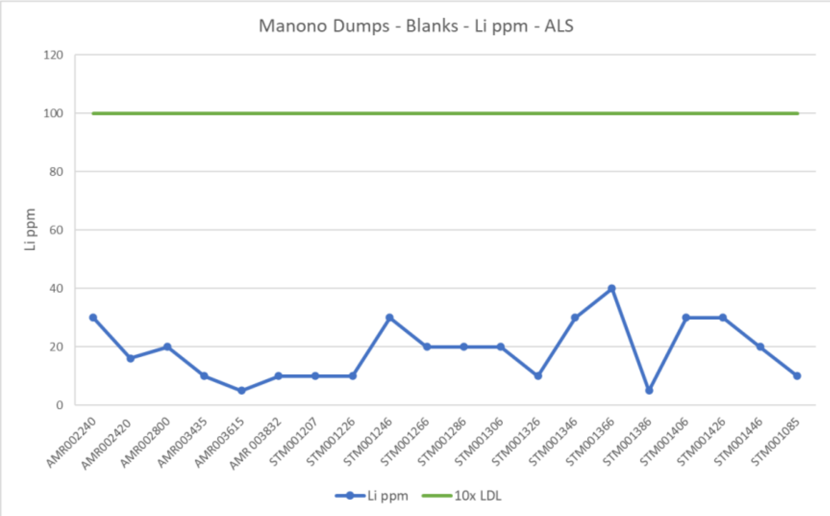
A summary of the number of blanks analysed and total failures is shown in Table 11-1.

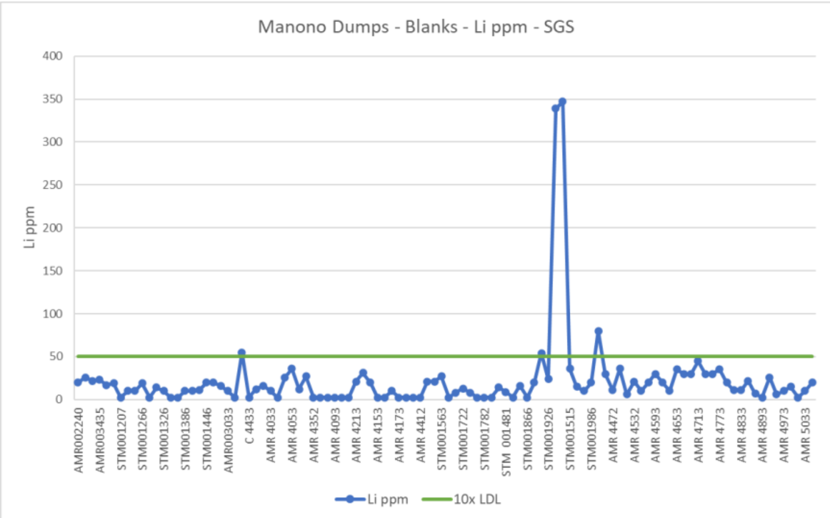
Table 11-1
Summary of blank samples used in the drilling programme

Li			Sn			Ta		
ALS	SGS	Failure Rate	ALS	SGS	Failure Rate	ALS	SGS	Failure Rate
20	112	4 %	3	87	1%	3	87	1%

The overall failure rate is low for the three elements. No failures were reported for lithium assays analysed by ALS, based on a threshold of 100 ppm which is ten times the lower detection limit (LDL). There are a total of five failures for lithium, two of which (AMR4713 and AMR4733) reported values well above the acceptable limit of 50 ppm, for SGS. Given that these failures are rare and isolated events and the degree of potential contamination is significantly below the lithium cut-off grade considered for the deposit, potential errors in this regard will not have a material impact to the Mineral Resource estimate. Graphical representations of the blank sample results for lithium are shown in Figure 11-7 for ALS and SGS.

Figure 11-7
2022 Li ppm in blank analyses (ALS and SGS)



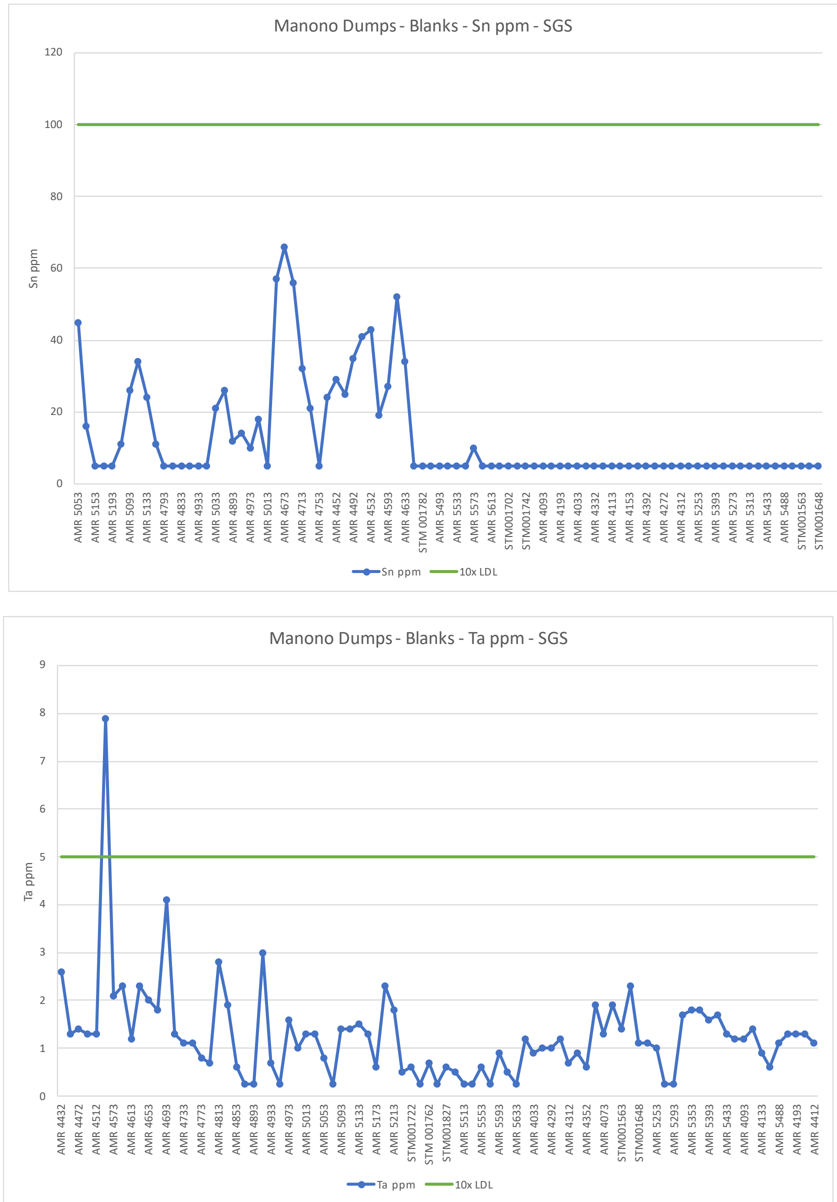


Source: *MSA (2022)*

A total of 90 blank samples were for analysed for Sn and Ta, with only three of these being analysed at ALS and the remainder at SGS. A ten times detection limit threshold was applied. The thresholds used to determine a failure for Sn and Ta were 100 ppm and 5 ppm respectively for samples analysed at SGS. The thresholds applied to the ALS samples were 10 ppm Sn and 1 ppm Ta. The overall failure rate for both elements is low, at 1% of the total samples, which equates to one failure for each element. The failure for Sn occurs on a sample analysed at ALS, while the failure for Ta was analysed at SGS.

Blank analysis control charts for Sn and Ta are shown in Figure 11-8. Graphs for ALS are omitted due to the small number of samples. The charts shows a decrease in tin grade reported in a blank sample after a change in analytical methodology was introduced by SGS following close monitoring of the results by Tantalex and MSA, which prompted an internal investigation by the laboratory. SGS adjusted the analytical method, by using a flame fusion with a hydrochloric acid digest instead of an oven fusion at 600°C and nitric acid digest.

Figure 11-8
Blank analyses for Sn and Ta (SGS only)



11.7.2 Certified Reference Material (CRM) Samples

CRM samples were purchased from AMIS and OREAS for insertion into the sampling stream at an approximate rate of 1 in every 20 samples. During the early stages of the drilling campaign, a lower rate of insertion was used which varied from 1 in 25 to 1 in 60 samples.

11.7.2.1 Lithium

Six different CRM samples were utilised with certified grades ranging from 1603 ppm Li to 7268 ppm Li. A summary of the number of CRMs for lithium, certified values, analytical failure rates and bias in terms of percentage and absolute differences is presented in Table 11-2.

Table 11-2
Manono Lithium Tailings Project certified CRM details for Li

CRM Name	Number of CRM samples	Certified Value (Li ppm)	Three Standard Deviations	Failure Rate		Difference	
				Number of Samples	Percentage of Failures	Average Bias	Absolute Difference (ppm)
AMIS0338	31	1707	477.0	1	3%	2%	37
AMIS0341	30	5041	333.0	1	3%	2%	103
AMIS0342	1	1603	298.5	0	0%	10%	181
AMIS0343	34	7180	2287.5	3	9%	3%	218
AMIS0355**	23	7268	1254.0	0	0%	4%	290
AMIS0629	32	2153	376.5	1	3%	1%	29
AMIS0656	-	-	-	-	-	-	-

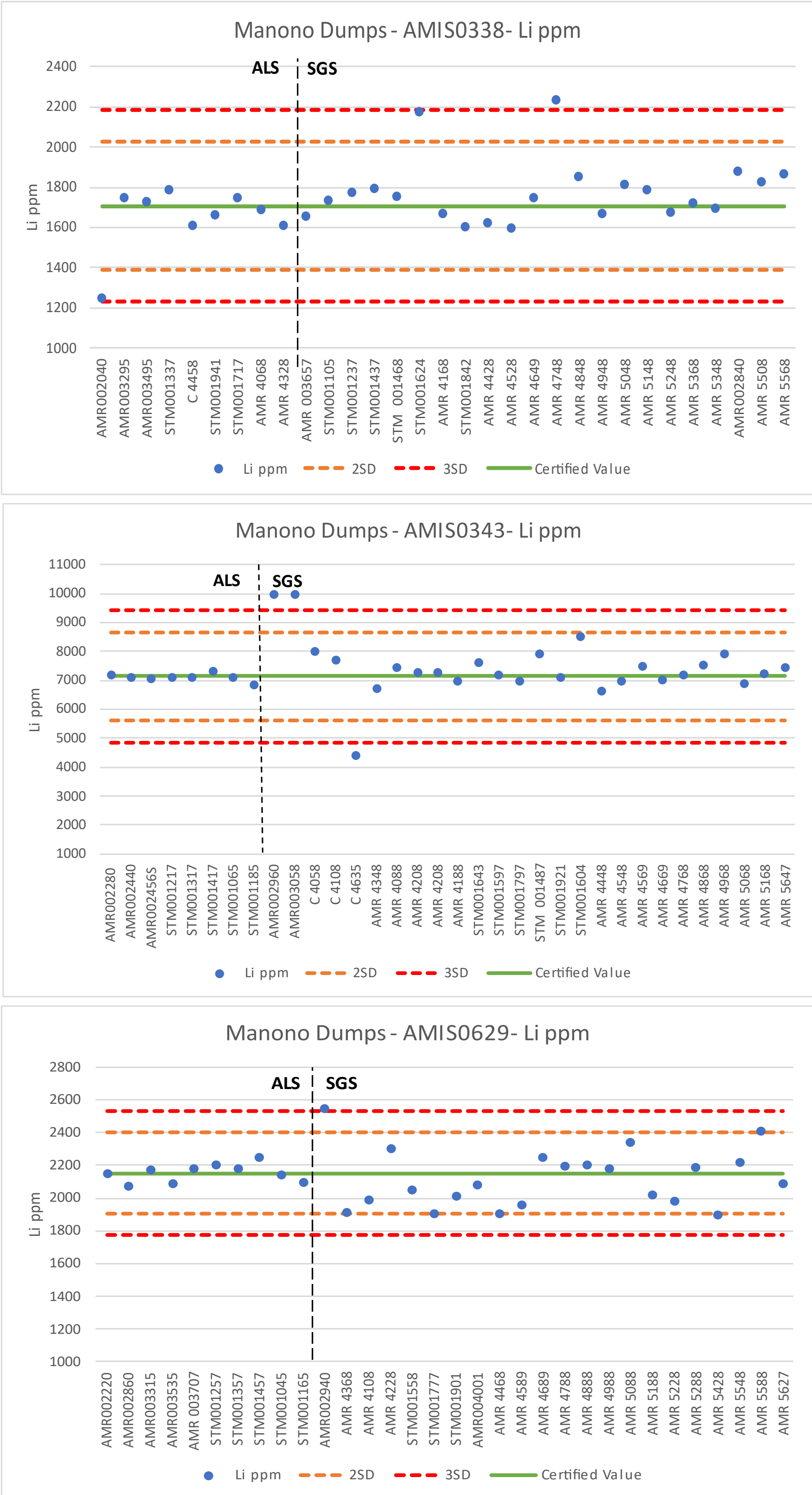
***Notes**: ** indicates ICP analysis*

AMIS0656, although certified for lithium, was only used to assess the accuracy of the tin and tantalum analyses. Two uncertified standards, WJL017 and WJL016, from Wheale Jane Laboratory in Cornwall, were inserted into the sampling stream during the earlier part of the resource drilling campaign. These were used as a temporary measure until certified CRMs were obtained. A total of 6 WJL016 and 5 WJL017 standard samples were used. All eleven were analysed for lithium while only nine were analysed for tin and tantalum. Due to their lack of certification, WJL standards were not used in the QAQC assessment.

There is a generally low (<4%) average bias between the analysed and certified values for lithium, except for AMIS0342 that reported 10% difference for a single sample that was analysed.

A selection of control charts representative of the lithium grade range of the CRMs assayed is presented in Figure 11-9. Most reported values are well within acceptable limits (three standard deviations of the certified value). Only one CRM assay (AMIS 0338) by ALS was a near failure (outside two standard deviations of the certified value), while one near failure was noted on the same CRM for SGS and one failure. Three failures were noted for AMIS0343 assayed at SGS, with two samples reporting >10000 ppm (i.e., above the upper detection limit for the analytical method) and one below the lower acceptance limit. Only one failure was noted for AMIS0629, which was an assay by SGS just outside of the upper acceptance limit.

Figure 11-9
Control charts for Li in CRMs AMIS0338, AMIS0343 and AMIS0629



***Source**: MSA (2023)*

11.7.2.2 Tin

The CRMs used in the drilling programme to assess the accuracy of tin assays have certified values ranging from 35.6 ppm to 6061 ppm. A summary of the number of CRMs for tin, certified values, analytical failure rates and bias in terms of percentage and absolute differences is presented in Table 11-3.

Table 11-3
Manono Lithium Tailings Project certified CRM details for Sn

CRM Name	Number of CRM samples	Certified Value (Sn ppm)	Three Standard Deviations	Failure Rate		Difference	
				Number of Samples	Percentage of Failures	Average Bias	Absolute Difference (ppm)
AMIS0338*	21	35.6*	10.5*	8	38%	18%	8
AMIS0342	1	1662	156	1	100%	-	-
AMIS0343	16	85	13.5	4	25%	8%	7
AMIS0355	16	470	57	0	0%	3%	13
AMIS0629	20	1662	156	0	0%	1%	17
AMIS0656	5	573	66	1	20%	24%	111
OREAS 140	3	1755	183	0	0%	3%	47
OREAS 141	1	6061	1017	0	0%	4%	251

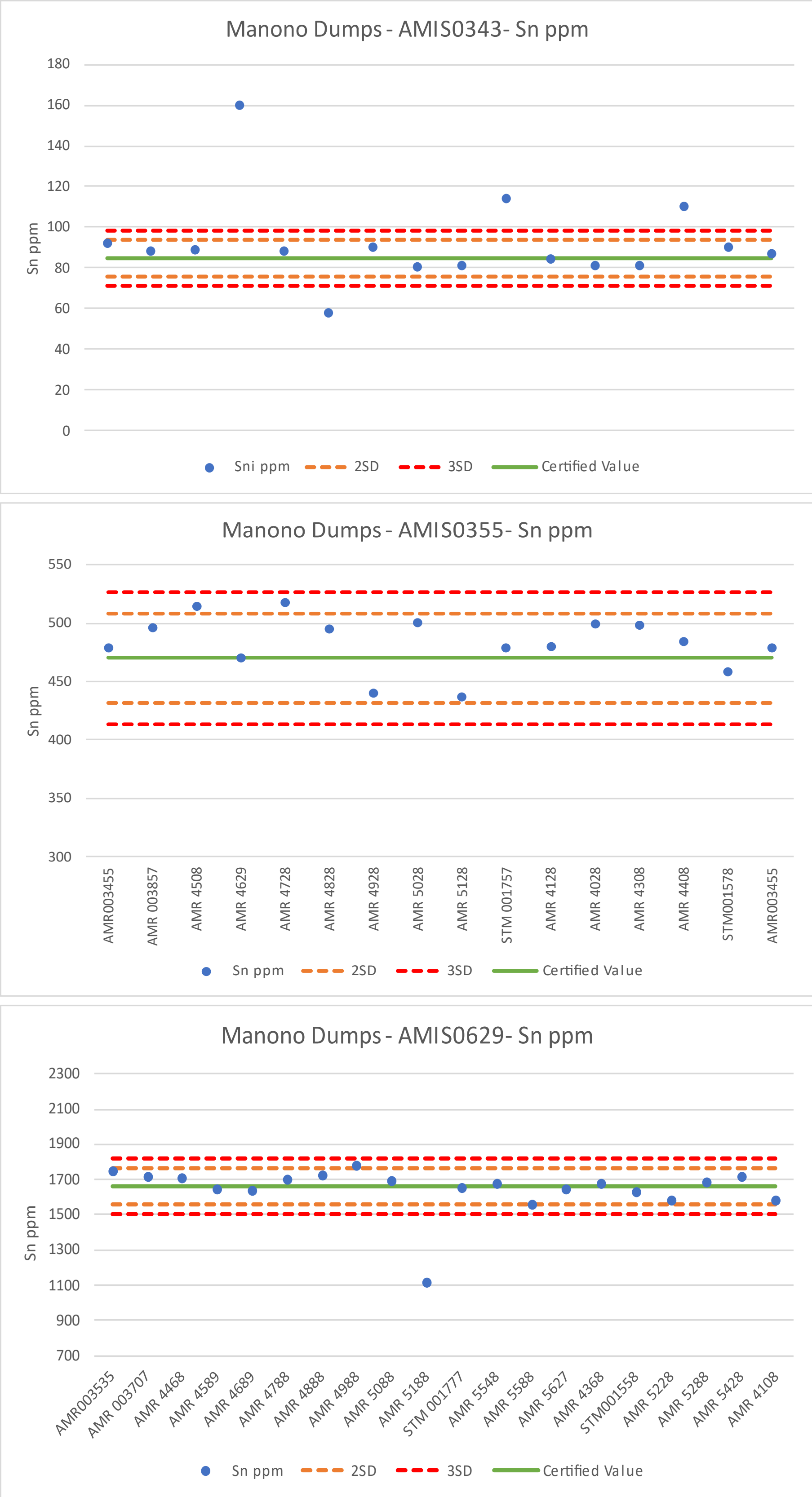
***Notes**: * indicates provisional values (not certified values)*

A small number of CRM samples were assayed for AMIS0342, AMIS0656, OREAS 140 and OREAS 141. These were too few to allow meaningful observations on the accuracy. A high number of failures, representing 38% out of a total of 21 CRMs were reported for AMIS0338, however, this standard is only provisionally certified for tin and therefore a conclusive opinion cannot be made. Twenty-five percent of the AMIS0343 samples failed by reporting outside the acceptable limits, although this CRM has a very narrow range of certification, with a three standard deviation value of 13.5 ppm. Regardless, the average bias and absolute difference for AMIS0343 is low. Similarly, low biases are noted for AMIS0355, and AMIS0629 which has a single failure below the certified mean.

A selection of control charts representative of the tin grade range of the CRMs assayed is presented in Figure 11-10.

Figure 11-10

Control charts for Sn in CRMs AMIS0343, AMIS0355 and AMIS0629



11.7.2.3 Tantalum

The CRMs used in the drilling programme to assess the accuracy of tantalum assays have certified values ranging from 43 ppm to 740 ppm. A summary of the number of CRMs for tantalum, certified values, analytical failure rates and bias in terms of percentage and absolute differences is presented in Table 11-4.

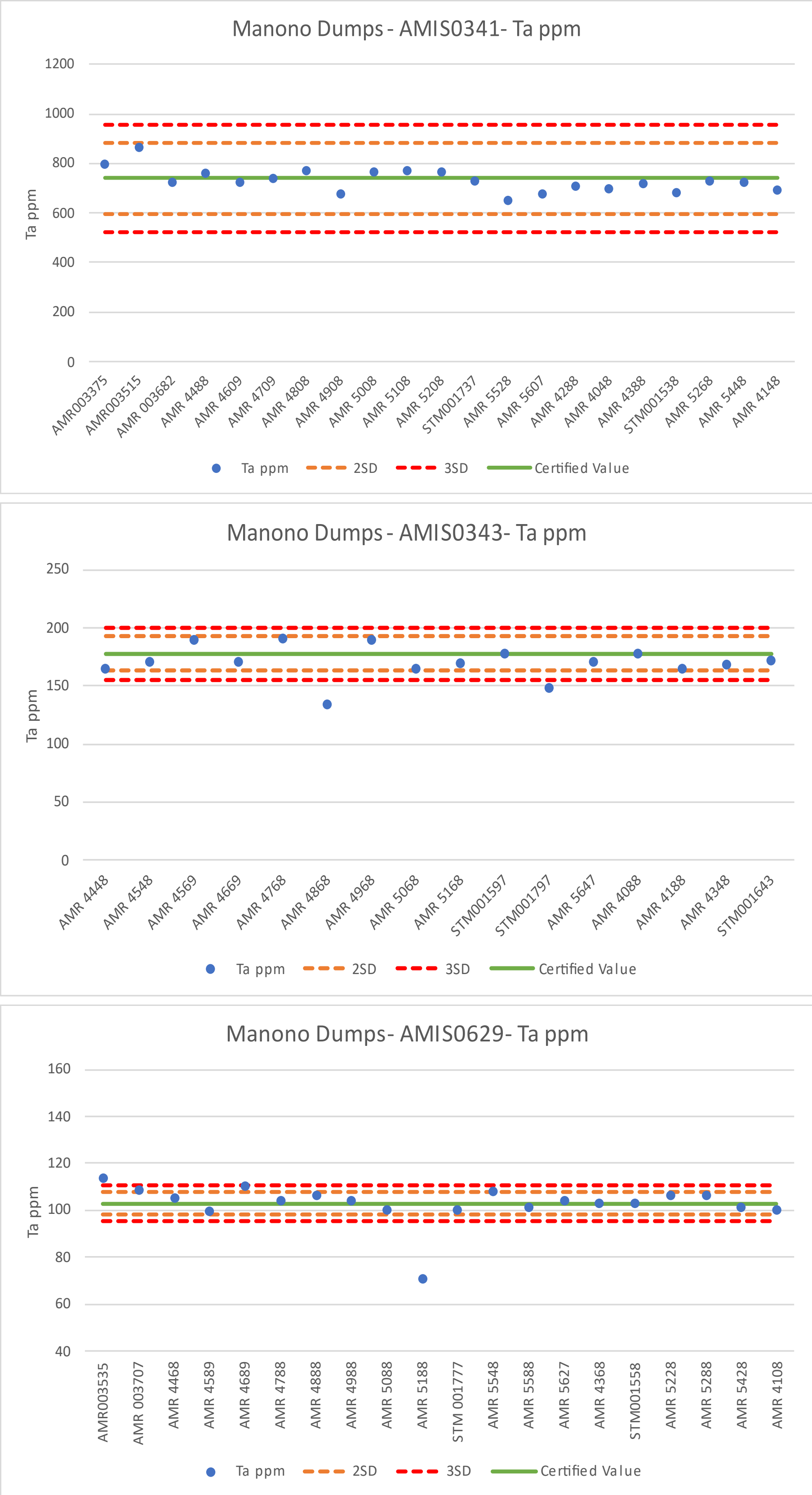
Table 11-4
Manono Lithium Tailings Project certified CRM details for Ta

CRM Name	Number of CRM samples	Certified Value (Ta ppm)	Three Standard Deviations	Failure Rate		Difference	
				Number of Samples	Percentage of Failures	Average Bias	Absolute Difference (ppm)
AMIS0338	21	43	15	1	5%	2%	1
AMIS0341	21	740	216	0	0%	1%	6
AMIS0342	1	169	25.5	0	0%	1%	1
AMIS0343	16	178	22.5	2	13%	5%	8
AMIS0355	16	214	63	0	0%	1%	3
AMIS0629	20	103	7.5	2	10%	0%	0
AMIS0656	5	179	39	1	20%	20%	29
OREAS 140	-	-	-	-	-	-	-
OREAS 141	-	-	-	-	-	-	-

Overall, the accuracy of the tantalum analyses is good, with low failure rates for most CRMs. One of the five AMIS0656 samples has an assay value significantly lower than the certified mean. AMIS0343 has a 13% failure rate, which represents 2 samples out of 16, while the average bias is low at 5% with an absolute difference of 8 ppm.

A selection of control charts representative of the tantalum grade range of the CRMs assayed is presented in Figure 11-11.

Figure 11-11
Control Charts for Ta in CRMs AMIS0341, AMIS0343 and AMIS0629

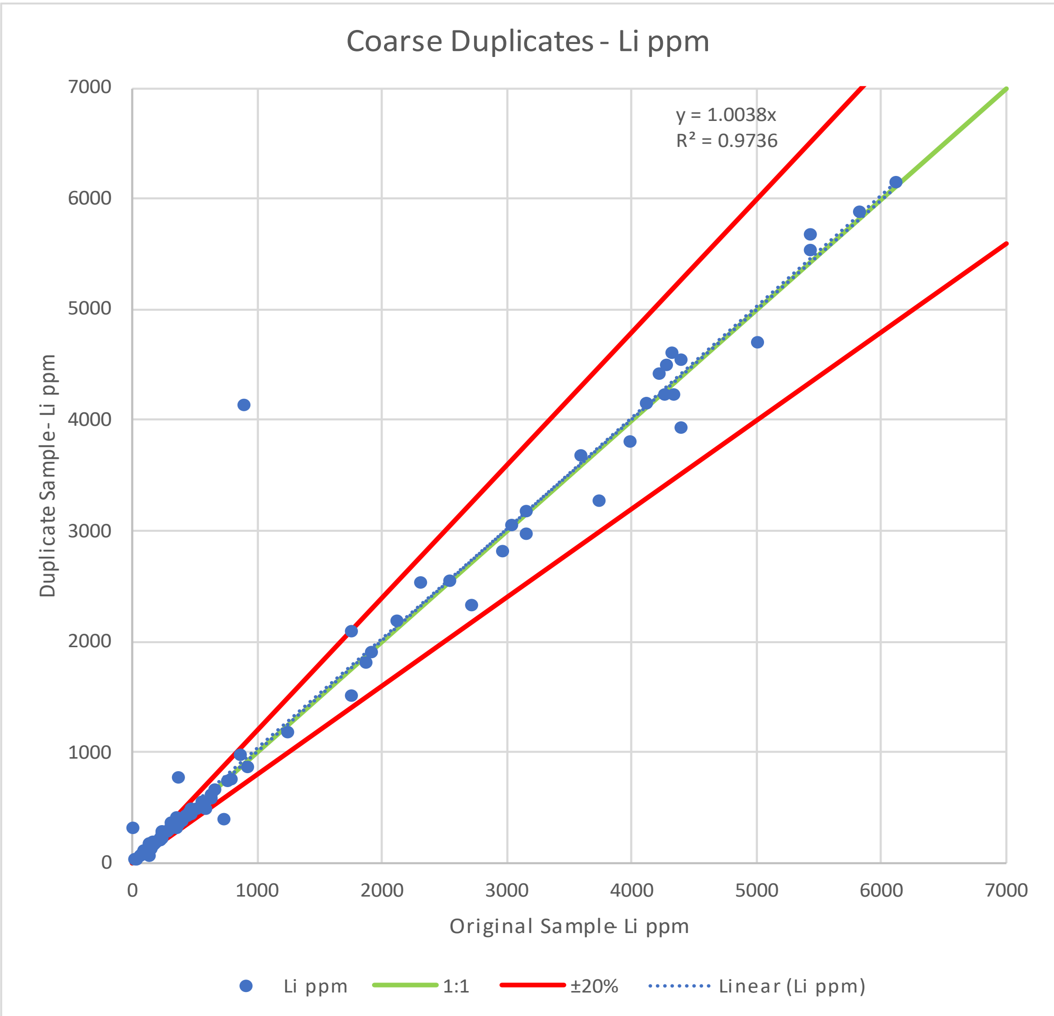


11.7.3 Duplicate Samples

11.7.3.1 Lithium

A total of 116 coarse duplicates were submitted by Tantalex for analyses, thirteen of these were submitted to ALS and the remainder to SGS. A comparison between the original and duplicate assays (Figure 11-12) for lithium shows good precision. This is corroborated by 89% of the samples having a half absolute relative difference (HARD) of less than 10% and 95% of the samples with a HARD of less than 20%. The mean lithium grade of the original samples is 1322 ppm compared to 1355 ppm for the duplicates. This small discrepancy in the means can be accounted for by a single anomalous sample pair that has a grade of 884 ppm Li for the original and 4130 ppm Li for the duplicate.

Figure 11-12
Precision of lithium in 108 coarse duplicate pairs

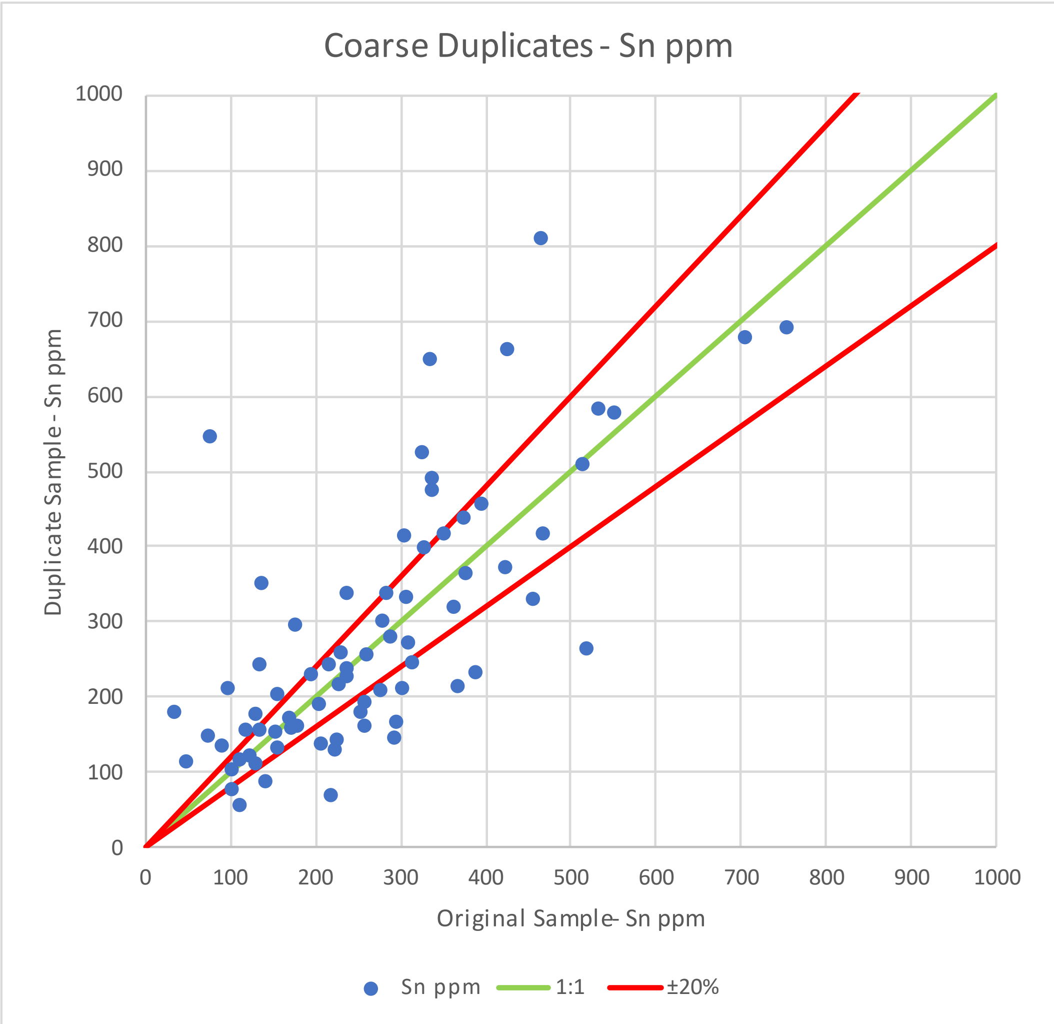


Source: *MSA (2023)*

11.7.3.2 Tin

A total of 78 coarse duplicates were submitted for tin analyses. Figure 11-13 shows a scatterplot comparing the tin assays of the original and duplicate sample pairs. The graph shows considerable scatter, and only 46% of the samples have a HARD value of less than 20%. However, in terms of mean values, the two legs are similar, with the original samples having a mean tin grade of 303 ppm versus 293 ppm for the duplicates.

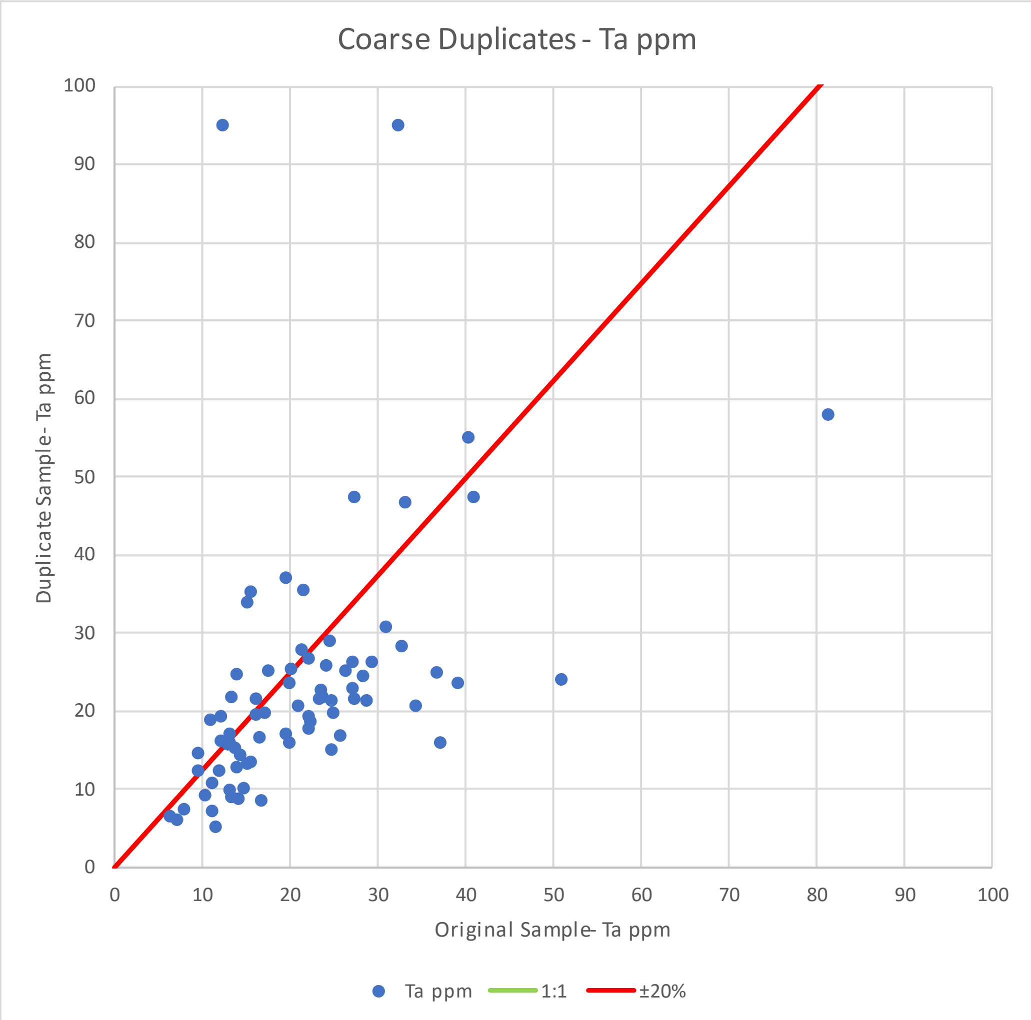
Figure 11-13
Precision of tin in 78 coarse duplicate pairs



11.7.3.3 Tantalum

A total of 78 sample pairs were analysed for tantalum. A scatterplot comparing the original and duplicate pairs is shown in Figure 11-14. This comparison shows poor precision with significant scatter and many pairs being greater than 20% different to each other. Similarly, as observed for tin, a total of only 46% of the tantalum assays have a HARD value of less than 20%. When comparing the means between the two datasets, the original samples have a mean tantalum grade of 21 ppm while the duplicate samples have a mean grade of 25 ppm. When excluding two outlier samples with HARD values above 100%, this difference becomes negligible.

Figure 11-14
Precision of tantalum in 78 coarse duplicate pairs



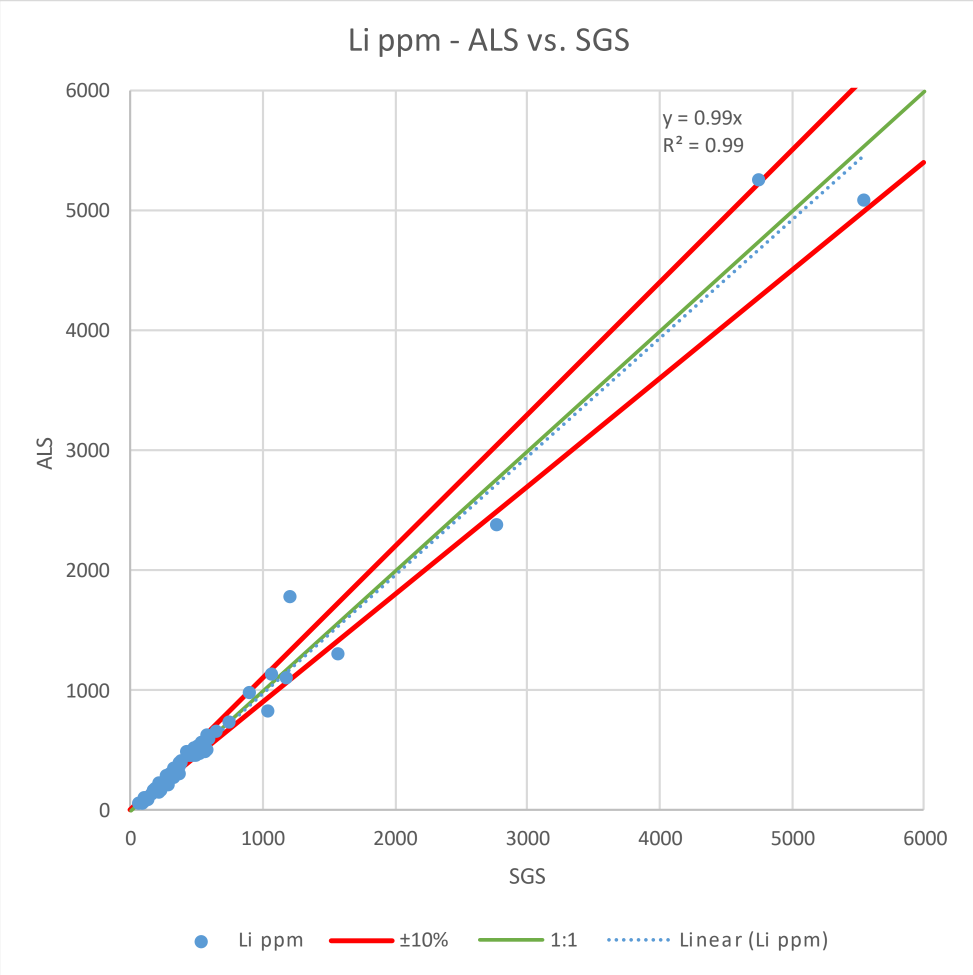
11.7.4 Second Laboratory Check Assays

Tantalex submitted a total of 67 samples for second laboratory check analysis in two batches. Of these, 66 sample pairs returned with a lithium assay. For tin and tantalum, only 40 sample pairs were used in the inter-laboratory comparison as the first batch was analysed at SGS before issues with the accuracy were identified.

11.7.4.1 Lithium

The correlation between SGS and ALS for lithium is presented in Figure 11-15, indicating good inter-lab precision, as evidenced by a R^2 value of 0.99 suggesting a strong linear relationship between the sample pairs.

Figure 11-15
ALS versus SGS – Li ppm



A summary on the sample repeatability between ALS and SGS for lithium is shown in Table 11-5.The mean of the ALS and SGS assays are comparable, with only a 1% difference and an absolute difference of 6.4 ppm. A total of 77% of the samples have a HARD of less than 10%, which is below the expected value of 90% for pulp samples. However, many of the samples pairs with poor correlation have grades ranging from 70 ppm to 200 ppm Li. At such low values, variability introduced during the sample preparation combined with equipment sensitivity and accuracy have a higher impact on analytical precision.

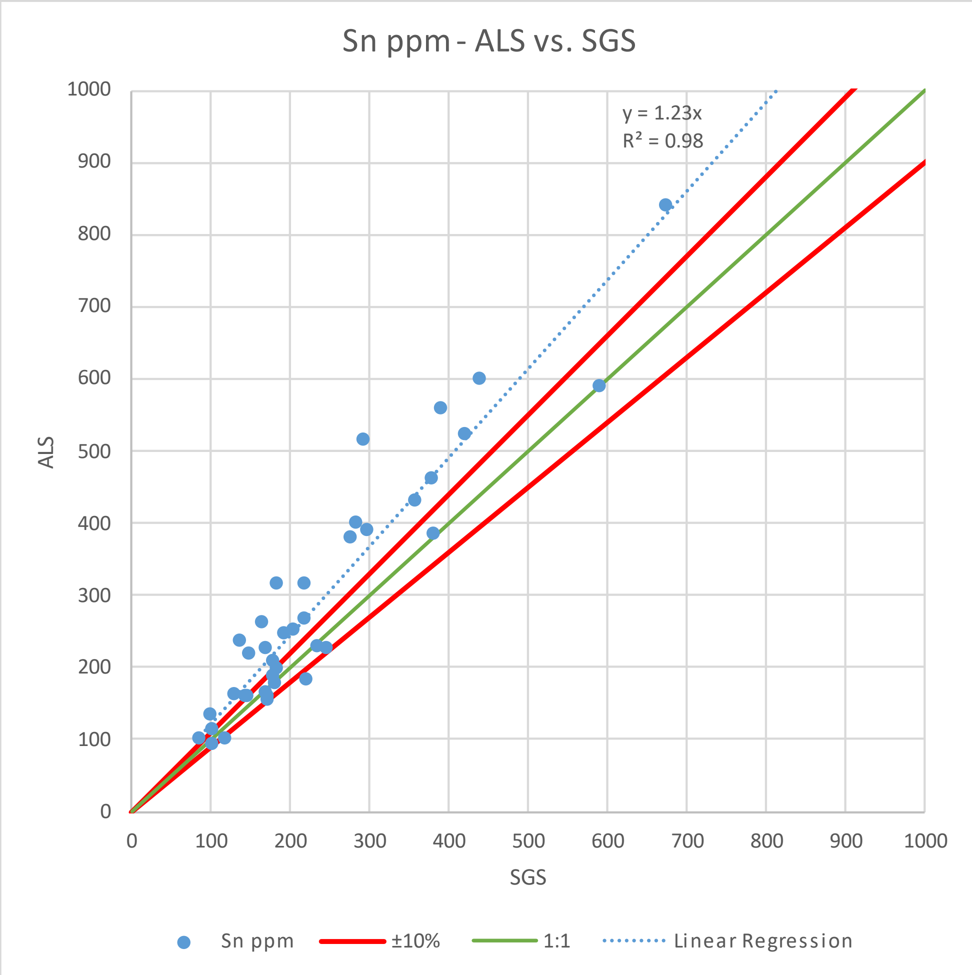
Table 11-5
Summary of sample repeatibility comparing ALS against SGS for lithium

Number of Samples	Mean ALS Li ppm	Mean SGS Li ppm	Percentage Difference	Absolute Difference Li ppm	HARD	
					<10%	<20%
66	597	603	1%	6.4	77%	97%

11.7.4.2 *Tin*

The correlation between the SGS and ALS sample pairs shows a strong bias towards ALS and significant scatter suggesting poor inter-lab precision for tin.

Figure 11-16
ALS versus SGS – Sn ppm



There is an 18% difference in the mean tin grade between ALS and SGS, which in absolute terms translates to a mean difference of 57.1 ppm Sn (Table 11-6). Only 40% of the samples have a HARD value of less than 10%, however 80% of the duplicate pairs have a HARD value of less than 20%.

Table 11-6 Summary of sample repeatability comparing ALS against SGS for tin						
Number of Samples	Mean ALS Sn ppm	Mean SGS Sn ppm	Percentage Difference	Absolute Difference Sn ppm	HARD	
					<10%	<20%
40	309	252	18%	57.1	40%	80%

11.7.4.3 Tantalum

Tantalum check assays show a slight bias towards ALS, although significant scatter is observed, particularly in grades below 25 ppm Ta (Figure 11-17).

Figure 11-17
ALS versus SGS – Ta ppm

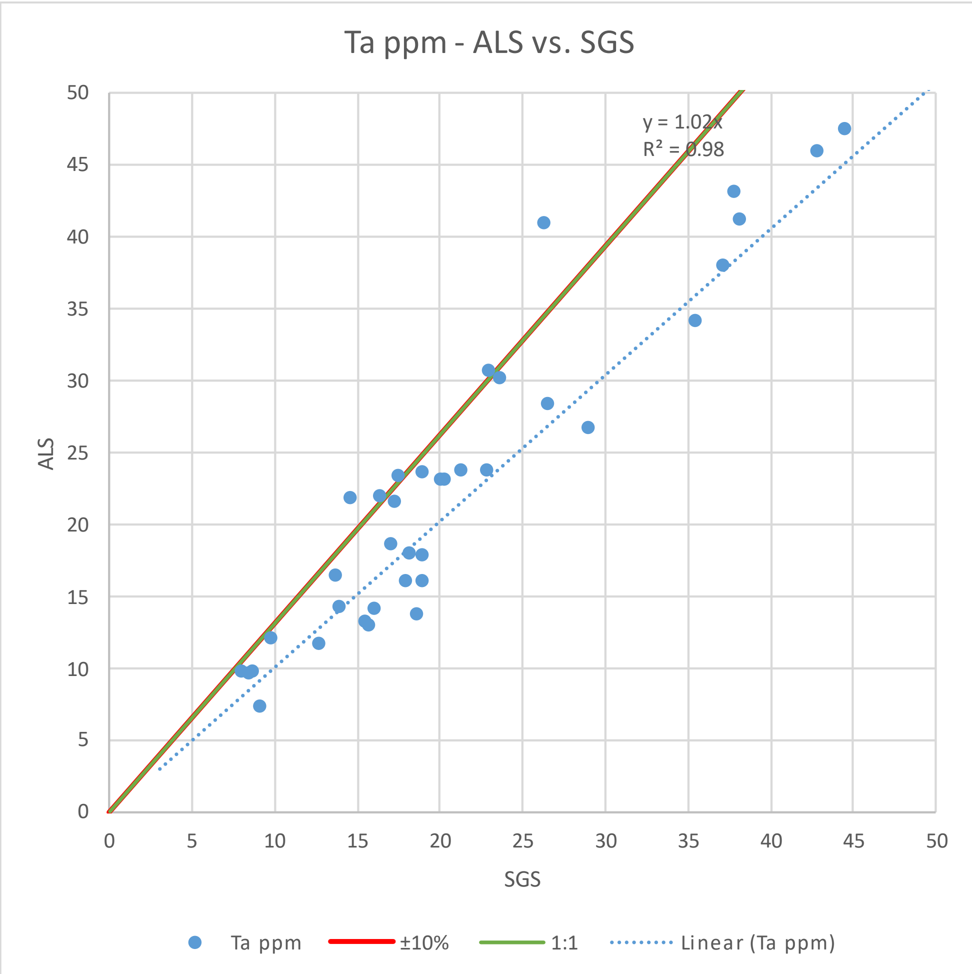


Table 11-7 shows that ALS assays are on average 6% higher than SGS, which in absolute terms translates to 2 ppm. Proportionally, 80% of the samples have a HARD less than 10%, which is below the ideal limits, although 93% of the tantalum assays have a HARD of 20% or less, which is acceptable.

Table 11-7
Summary of sample repeatability comparing ALS against SGS for tantalum

Number of Samples	Mean ALS Ta ppm	Mean SGS Ta ppm	Percentage Difference	Absolute Difference Ta ppm	HARD	
					<10%	<20%
40	28	26	6%	2	80%	93%

It is likely that sample heterogeneity at low grades will impact precision for tantalum.

11.8 Density Measurements

In 2022, a sampling program was conducted by Tantalex to support a dry bulk density calculation for use in the Mineral Resource estimate. Samples were collected from 64 sample locations on five coarse tailings dumps (and associated fine tailings terraces), namely "G", "H", "I", "K" and "C" dumps. Lithologies sampled included pegmatite, laterite and clay (Kinyaga, 2022).

A pit was excavated approximately 1 m below surface in order to avoid sampling less consolidated tailings at surface. Density samples were collected by driving a steel cylinder into the pit base with the assistance of an excavator. The steel cylinder was dug out with shovels to prevent any loss of the contained tailings material (Figure 11-18). The cylinder contents were transferred to a sampling bag and the weight (wet and dry) was recorded. Density was calculated using the formula:

$$Density = Weight_{(dry)} \,/\, Volume_{(cylinder)}$$

Two samples, approximately 3 m to 4 m apart, were averaged to calculate the density for one excavation.

Figure 11-18
Bulk density sampling

***Source**: Adapted from Kinyaga (2022)*

The results of the dry bulk sampling program indicate different densities are applicable to different lithologies. Average density was assigned in the Mineral Resource estimate for each deposit based on the density data belonging to that particular tailings. A summary of the density data is presented in Table 14-13.

Table 11-8
Density ranges and averages per material type

Material Type	Number of Samples	Minimum	Maximum	Mean
Laterite	22	1.42	1.77	1.65
Clay	6	1.13	1.45	1.29
Metasediment	4	1.52	1.68	1.61
Pegmatite	86	1.35	1.78	1.57
Pegmatite Sand	8	1.42	1.56	1.49
Pegmatite Clay	2	1.44	1.56	1.56

11.9 Adequacy of Drilling Procedures, Sample Preparation, and Analytical Procedures

All aspects of the sample handling, logging, bagging, labelling and sample submission processes are considered reasonable and acceptable for use in a Mineral Resource estimate. MSA recommends that Tantalex develops in-house Standard Operating Procedures (SOPs) for any future drilling programs that will cover geological logging, drilling, sampling, QAQC, sample storage and data management.

The analyses of the QAQC data found the following:

- There is no indication of significant contamination for lithium, tin and tantalum, with blank samples reporting low failure rates.
- The CRM analysis for lithium, tin and tantalum show an acceptable level of accuracy. The number of failures is generally low for each element and the average bias between the samples and certified values is often insignificant.
- Coarse duplicates for lithium indicate good precision, with little bias between the original and duplicate sample pairs. However, analytical precision is poor for both tin and tantalum.
- Second laboratory duplicate checks on lithium by ALS largely confirm the SGS results, although a slight bias was noted towards the primary laboratory.

The re-assay exercise for tin and tantalum resulted in a vast improvement on the accuracy of the analyses although precision remained poor at the low grades of the tailings material.

MSA considers that the lithium assays from the 2021-2022 drilling program are of acceptable quality for use in a Mineral Resource estimate as demonstrated by the QAQC data. Although analytical precision is poor, the tin and tantalum assays are of acceptable accuracy and there is no indication of contamination. Repeatability can be poor with low grade tin and tantalum samples due to the nuggety nature of the cassiterite and tantalite mineralisation even after milling. Poor precision impacts on local selectivity and enough samples should be used in the estimation to cater for high grade variability caused by poor precision.

12 DATA VERIFICATION

A "Current Personal Inspection" was conducted by the Qualified Person for the Mineral Resource on 29 and 30 April 2022.

- No drilling activities were taking place at the time of the site visit. The first phase of the drilling campaign ended in November 2021, with the second phase beginning on 15 June 2022, after the site visit took place.
- An inspection of K, Gc, Hf and Hc deposits was undertaken. The tailings deposits were observed to align with the topographical surveys generated by Tantalex.
- The collars of 16 Tantalex drillholes were located and the collar coordinates were taken with a handheld GPS. The final surveys of the collar positions correlated reasonably well with the measurements taken with the handheld GPS, within acceptable limits for handheld GPS measurements (Table 12-1).

Table 12-1
Comparison between surveyed coordinates and handheld GPS measurements for a selection of drillhole collars

Drillhole ID	Collar Coordinates		GPS Coordinates		Difference (m)	
	X	Y	X	Y	X	Y
MDA001	545070.0	9190869.9	545073.0	9190864.5	-3.0	5.4
MDA002	545012.0	9190921.9	545015.3	9190916.1	-3.3	5.8
MDA048	543780.9	9190539.1	543784.3	9190534.5	-3.4	4.5
MDA050	543721.3	9190556.7	543722.9	9190551.7	-1.6	5.0
MDA051	543730.6	9190533.1	543730.3	9190528.5	0.4	4.5
MDA054	543686.8	9190550.3	543687.4	9190545.5	-0.7	4.8
MDA059	543636.0	9190701.6	543635.9	9190697.4	0.1	4.2
MDA061	543587.8	9190638.4	543589.6	9190634.1	-1.9	4.3
MDA062	543569.1	9190610.9	543570.1	9190606.0	-1.0	4.8
MDA067	543587.7	9190578.4	543591.4	9190574.6	-3.7	3.8
MDA068	543584.0	9190541.6	543586.6	9190538.5	-2.6	3.1
MDA100	542157.1	9189247.0	542157.1	9189242.0	0.1	5.0
MDA101	542164.9	9189084.7	542163.5	9189080.0	1.4	4.7
MDA102	542015.9	9189210.7	542016.9	9189205.6	-0.9	5.1
MDA103	542293.4	9189374.6	542296.1	9189369.3	-2.7	5.3
MDA104	542225.2	9189514.1	542226.2	9189510.2	-1.0	3.8

- The original paper logs were inspected. These are in good condition and stored in a secure location in Manono.
- The chip trays for a selection of the completed drillholes were inspected, including the five drillholes that were available for the K dump, namely MDA100, MDA101, MDA102, MDA103 and MDA104. The logging was found to be an accurate representation of the material contained in the chip trays. The mineralisation observed in the chips was compared with

the assay data available at the time and some high lithium grades could be correlated with identifiable spodumene mineralisation.

- The logging and sampling procedures were discussed with Tantalex geologists on site and these were found to be appropriate for the purpose of evaluating the Mineral Resource.

12.1 Check Sampling

As part of a data verification exercise, 16 samples from the K dump were selected from the available reject samples stored at the sample preparation facility and re-submitted to SGS Johannesburg for analysis. The samples were sealed by the QP using numbered tamper proof cable ties in order to ensure that these were not tampered with. Confirmation of the intact, tamper-proof sealed samples was received by Mr. Jhoel Mbuya of COAL on 04 June 2022 (Figure 12-1).

Figure 12-1
Sealed check samples from Manono at COAL

Source: *Mbuya (2022)*

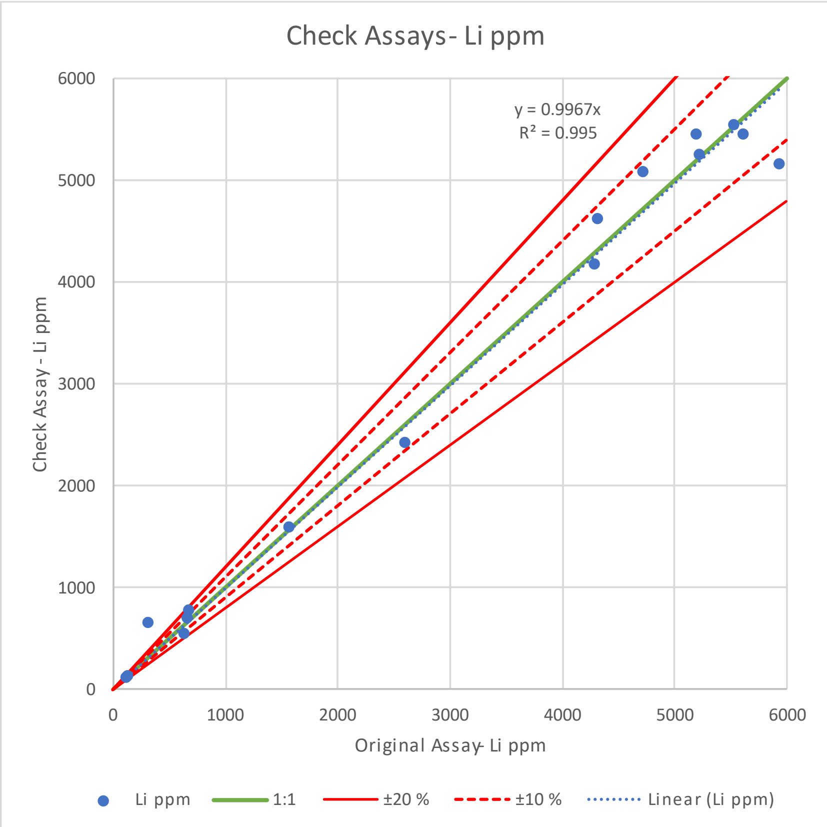
The samples underwent the same sample preparation and analytical procedure at SGS South Africa as the original Tantalex samples. The check samples were done on the first sample submission to SGS, prior to the issues with the tin and tantalum being identified, therefore the results for these two elements were discarded. As the pulps for these samples were no longer available for re-analysis for tin and tantalum, only lithium results were considered for the check samples.

A statistical comparison between the original and check samples for lithium is shown in Table 12-2. The mean lithium grades of the original and check assays are consistent, with a 0.9% difference between the mean values of the two sets of data and a similar coefficient of variation (CV). Only one sample pair was outside the 20% limits and therefore the check assays confirm the original lithium assays.

Table 12-2 Comparison of original with the check assays						
Attribute	**Original Assay**		**Check Assay**		**Percentage Difference**	**Percentage Outside 20% Limit**
	Mean	**CV**	**Mean**	**CV**		
Li ppm	2967	0.76	2995	0.73	0.9%	6%

Scattergrams were used to compare the check assays with the original assays. Figure 12-2 illustrates the high correlation and minimum bias between the two sets of lithium assay data.

Figure 12-2
Scattergram for lithium – check vs. original samples



12.1.1 Qualified Persons opinion on the check assaying

The check samples for lithium have confirmed the original sample analyses, with no significant bias identified. Similar checks on tin and tantalum were not possible, as the original samples were analysed prior to issues with the SGS analytical procedure were identified, with sample pulps no longer being available.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The metallurgical testwork on the Project was carried out in 2022 and 2023 using bulk samples obtained from C-dump, G-dump, and K-dump. The samples were subjected to testing at several laboratories as summarized in Table 13-1.

Table 13-1
Bulk Sample Tests and Laboratories

Laboratory	Test	Location
Coremet Mineral Processing	• Feed Grade Mineralogy • Beneficiation • Granulometry • Crushability • Heavy Liquid Separation • Fines Beneficiation	Pretoria, South Africa
Pesco Services	Dense Media Separation	Durban, South Africa
Sepro Laboratories	Dense Media Separation	Vancouver, Canada
SGS Canada	Flotation	Lakefield, Canada
Nagrom	Reflux Classifier	Perth, Australia

The purpose of this testwork was to:

a) Establish the mineralogical characterisation of the dump material and determine if there were any significant variabilities across the different dumps.

b) Provide information of all the valuable minerals contained in the tailings dumps and develop adequate beneficiation methods.

c) Determine the crushability of the dumps and select a crushing top size for further processing.

d) Perform a Dense Media Separation (DMS) investigation on the coarse material fraction to determine the ideal cut points for maximum Li2O recovery.

e) Perform Flotation testwork to determine the ideal parameters for Li2O recovery.

f) Perform Reflux Classifier (RC) testwork to determine this technologies affinity for mica removal.

g) Perform slimes beneficiation testwork to determine the recovery potential of heavy minerals (Sn and Ta).

All available results at the time of this report are presented here, with additional results expected by Q3 2023.

13.2 Testwork Sample Selection and Feed Grades

To conduct the mineral characterisation testwork, bulk sample locations were selected based on the assay results from the aircore and cobra drillholes shown in Table 13-2. A total sample of 9,015 kg was collected of which 7,964 kg of sample was shipped and received by CoreMet in South Africa. The bulk samples are considered to be representative of the type and style of mineralisation of the deposit.

Table 13-2
Bulk sample locations and weights

Prospect	Hole ID Position Sample	Lithology	Weight (kg)	+25mm (kg)	-25 mm (kg)	% +25mm	% -25mm	Sample +25mm (kg)	Sample -25mm (kg)	Total Sample (kg)
K-dump	MDC047	Pegmatite	4,860	0	4,860	0	100	0	1,002	1,002
K-dump	MDC056	Pegmatite	4,681	0	4,681	0	100	0	1,008	1,008
K-dump	MDC064	Pegmatite	4,860	0	4,860	0	100	0	1,008	1,008
G-dump	MDA048	Pegmatite	9,952	843	9,109	8	92	127	1,373	1,500
G-dump	MDA059	Pegmatite	3,940	94	3,846	2	98	36	1,464	1,500
C-dump	MDA150	Pegmatite	5,067	104	4,963	2	98	31	1,469	1,499
C-dump	MDA158	Pegmatite	6,317	1,388	4,929	22	78	329	1,170	1,499
		Total	**39,677**	**2,430**	**37,247**	**6**	**94**	**522**	**8,493**	**9,015**

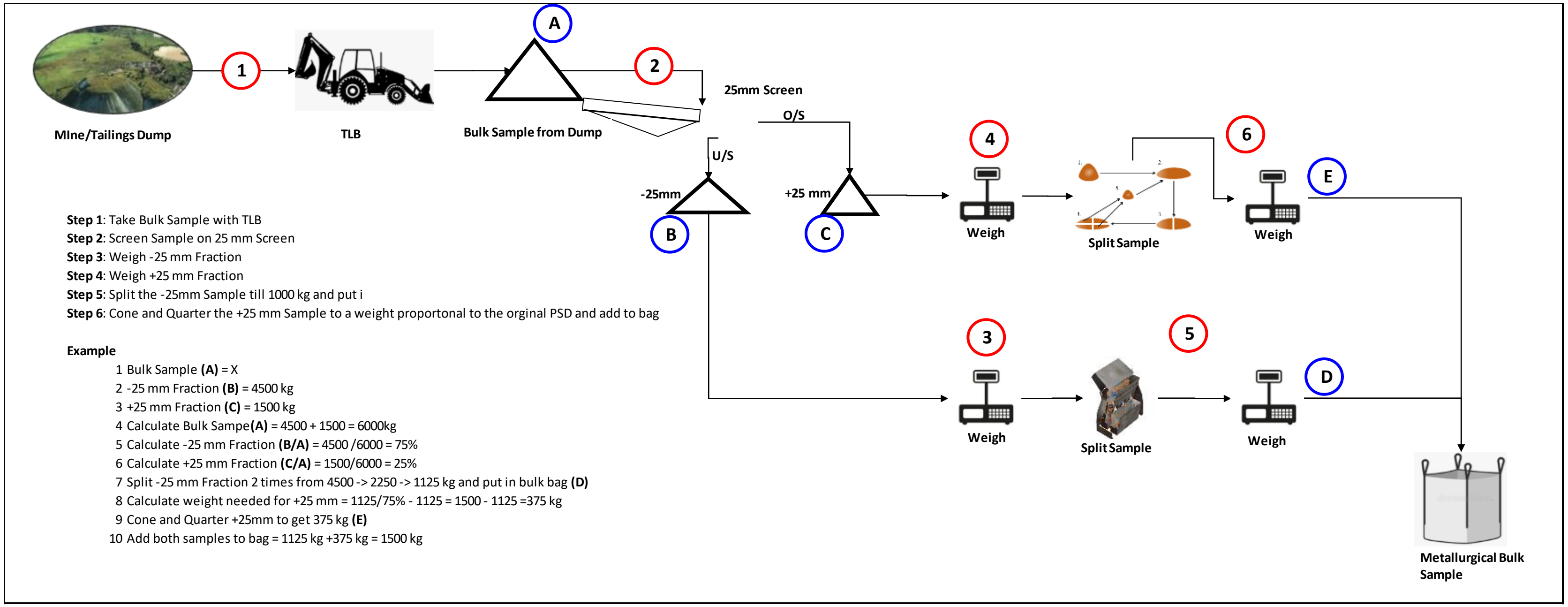
The lithium, tin and tantalum feed grades associated with each dump are tabulated in Table 13-3. Tin and tantalum grades across the dumps are similar while lithium grades are more variable.

Table 13-3
Feed Grades

Dump	Li_2O %	Sn ppm	Ta ppm
C-Dump	0.33	443.35	38.16
G-Dump	0.61	464.42	32.17
K-Dump	1.05	485.55	34.08

The philosophy of the bulk sampling process is illustrated in Figure 13-1.

Figure 13-1
Bulk Sampling Process



13.3 Mineralogical Testwork

Prepared samples from each dump were sent for chemical analysis by Inductively Coupled Plasma ("ICP") and X-Ray Diffraction ("XRD") to understand the composition of each dump as well as the distribution of target minerals in the various dumps.

The mineralogical analysis for the feed samples is tabulated in Table 13-4. The mineralogical analysis of the three dumps with associated HLS testwork indicated the following:

- Nearly all the lithium is contained in spodumene,
- Cassiterite is the only tin bearing mineral,
- The majority of tantalum occurs as tantalite with low concentrations of tapiolite,
- The main gangue minerals identified were quartz (25-37%), albite (18-38%), Microline (12-21%) and muscovite (4-15%).

Table 13-4
Feed Sample Mineralogical Analysis

Mineral	Empirical Formula	Density (t/m³)	C-Dump	G-Dump	K-Dump
Albite	$NaAlSi_3O_8$	2.62	18.24	37.49	37.95
Clinochlore	$(Mg,Fe^{2+})_5Al(Si_3\ Al)O_{10}(OH)_8$	2.65	2.79	0.00	1.07
Kaolinite	$Al_2Si_2O_5(OH)_4$	2.60	0.00	0.00	0.00
Magnetite	Fe_3O_4	5.15	2.52	3.29	1.35
Microline	$K(AlSi_3O_8)$	2.57	21.06	11.98	13.91
Muscovite	$KAl_2(Si_3Al)O_{10}(OH,F)2$	2.80	14.60	3.89	7.32
Quartz	SiO_2	2.62	36.94	35.80	25.56
Spodumene	$LiAlSi_2O_6$	3.15	3.83	7.53	12.85
Total			**100**	**100**	**100**

13.4 Beneficiation Testwork

Particle size analysis ("PSD") and heavy liquid separation ("HLS") testwork were conducted to establish that the lithium, tin, and tantalum can be extracted from the dumps.

The HLS tests were performed over four size fractions, -5 mm +1.2 mm, -1.2 mm +0.6 mm, -0.6 mm +250 µm and -250 µm +45 µm. The +5 mm fraction was not tested during this phase of testwork. The heavy liquid separation testwork was performed over three densities which included 2.95, 3.20 and 3.50 t/m^3. Each density produced a float and sink stream that was subject to mineralogical and elemental analysis to determine recoveries and grades.

The as received bulk samples were crushed to 100% passing 15 mm and screened at 5 mm. Figure 2-1 shows that K-dump has very limited +5 mm particles which will limit crushing requirements. C-dump and G – dump have a similar PSD distribution with about 30% of the material subject to further crushing if it proves to be economical.

Figure 13-2
Particle size distirbution

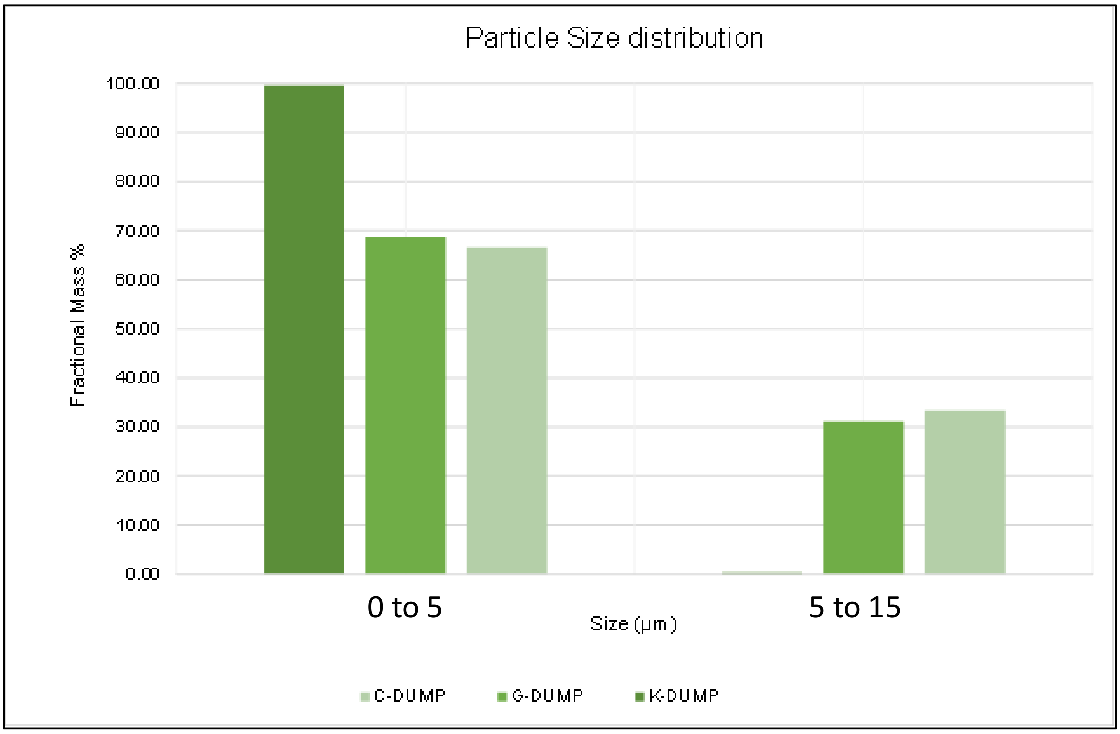
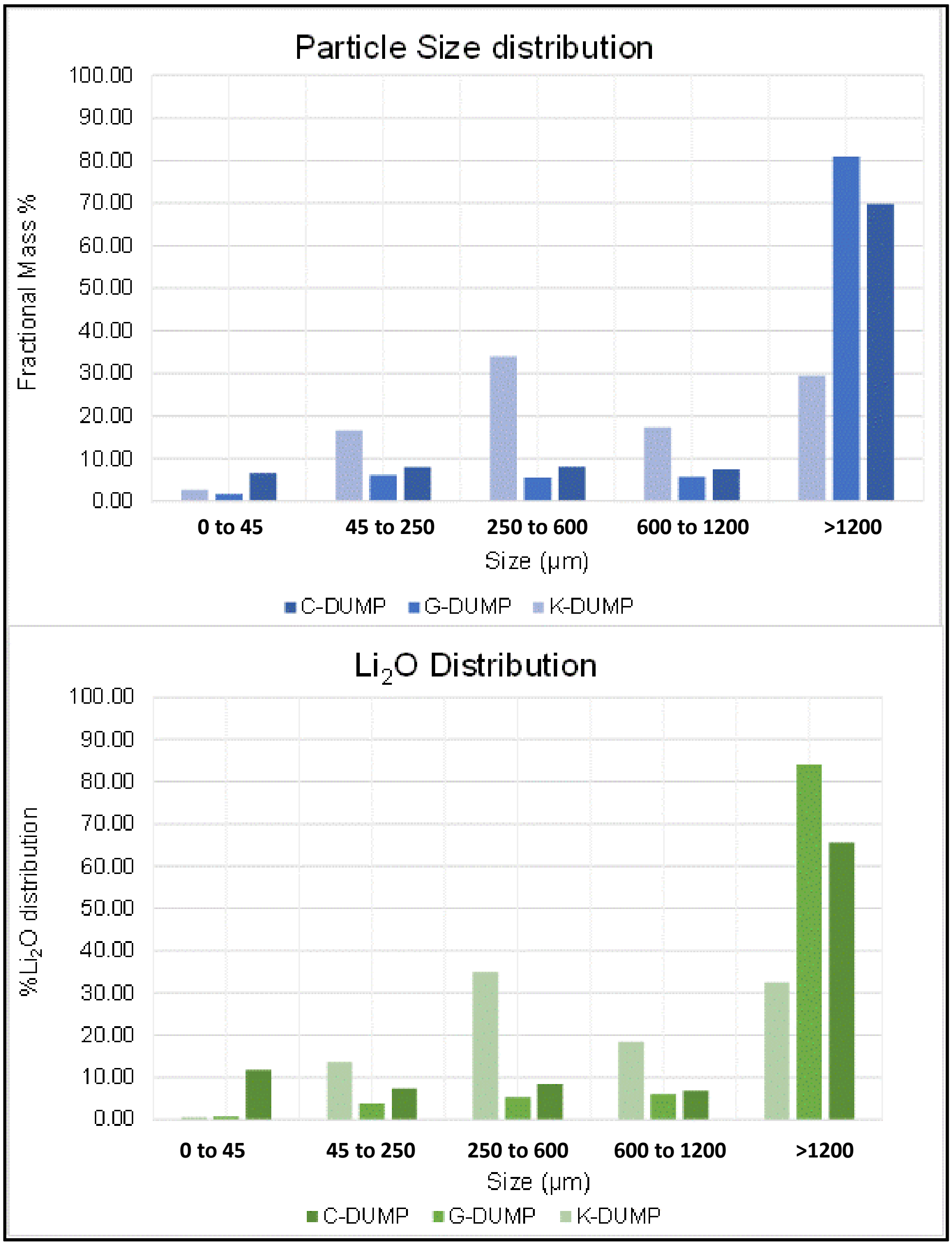


Figure 13-3 shows the PSD and lithium distribution across the dumps that were subjected to the HLS testwork. It can be concluded that lithium is evenly distributed across the different size range for K-dump while it is concentrated in the -5 mm +1.2 mm fraction for C-dump and G-dump. This means that most conventional mineral processing techniques can be utilised to process the coarse fraction (+0.5 mm). At size fraction less than 0.5 mm conventional beneficiation methods have inferior efficiencies and more site specific methods will have to be investigated to optimise the recovery in the fine fraction.

Figure 13-3
HLS feed PSD and Distirbution



A summary of the HLS results is available in Table 13-5. The HLS test produced spodumene concentrate grades of 6.5% Li_2O at overall recoveries across the size range of 47% and 63% for G-dump and K-dump respectively. The testwork did not produce a SC6 product from the C-dump, this requiring further investigation. These results are for all the dump material with a PSD smaller than 5 mm.

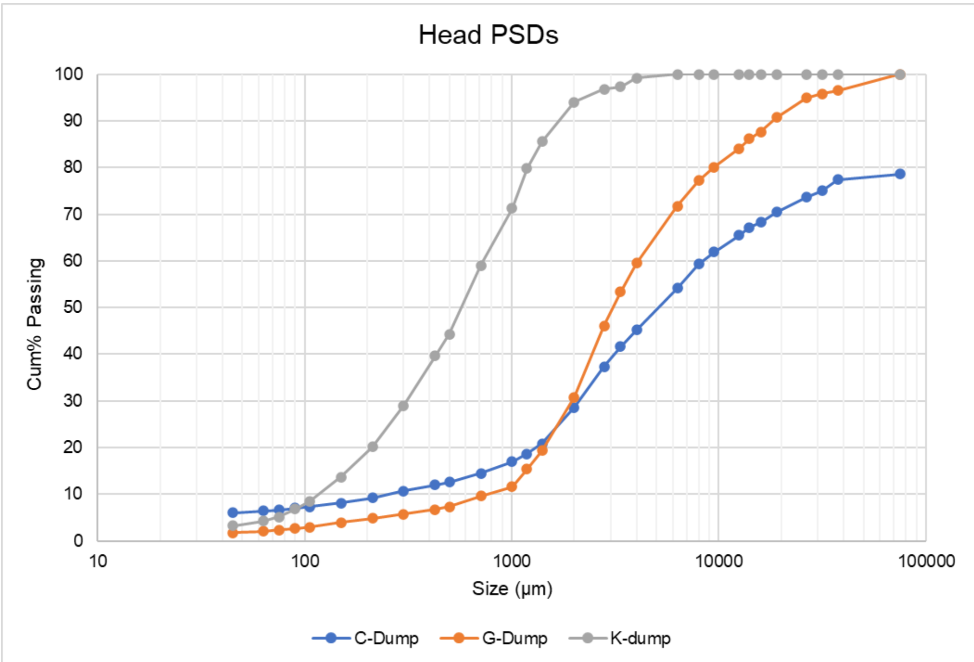
The lithium recoveries increased with size fraction while the tin and tantalum required further liberation to improve recoveries.

Table 13-5 HLS Summary Results					
Element	**Item**	**Unit**	**C-Dump**	**G-Dump**	**K-Dump**
Li_2O	Head Grade	%	0.33	0.61	1.05
	Recovery	%	28	47	63
	Concentrate Grade	**%**	**4.9**	**6.5**	**6.5**
Sn	Head Grade	ppm	443	464	486
Ta	Recovery	%	34	41	24

13.5 Granulometry

Coremet conducted head grade PSD analysis of C-dump, G-dump, and K-dump and is presented in Figure 13-4.

Figure 13-4
K-dump, G-dump and C-dump PSDs

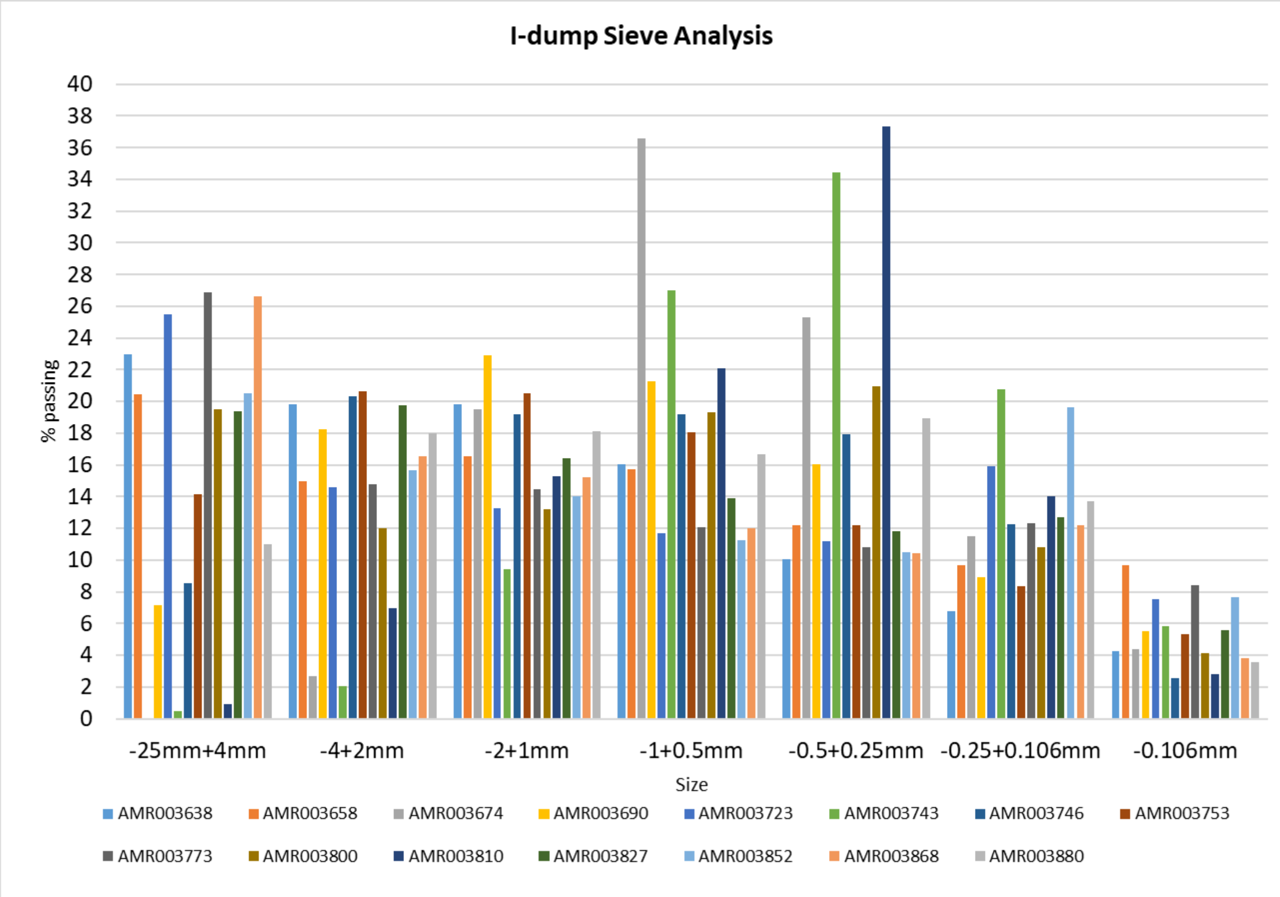


Tantalex performed a sieve analysis on four I-dump boreholes at multiple depths and the results are presented in Table 13-6 and Figure 13-5. These preliminary results indicate that G-dump and I-dump have similar granulometry and it has been assumed that they would behave similarly during processing.

Table 13-6
I-dump Sieve Analysis Results

Hole_ID	Sample_ID	From (m)	To (m)	-25mm+4mm	-4+2mm	-2+1mm	-1+0.5mm	-0.5+0.25n	-0.25+0.10	-0.106mm
MDA083	AMR003638	5	6	23.0	19.8	19.8	16.0	10.0	6.8	4.3
MDA083	AMR003658	24	25	20.4	15.0	16.5	15.7	12.2	9.7	9.7
MDA083	AMR003674	40	41	0.0	2.7	19.5	36.5	25.3	11.5	4.4
MDA087	AMR003690	11	12	7.1	18.2	22.9	21.3	16.0	8.9	5.5
MDA087	AMR003723	43	44	25.5	14.6	13.3	11.7	11.2	15.9	7.5
MDA087	AMR003743	62	63	0.5	2.0	9.4	27.0	34.5	20.7	5.8
MDA087	AMR003746	65	66	8.6	20.3	19.2	19.2	17.9	12.2	2.5
MDA093	AMR003753	4	5	14.2	20.6	20.5	18.1	12.2	8.4	5.3
MDA093	AMR003773	23	24	26.9	14.8	14.5	12.0	10.8	12.3	8.4
MDA093	AMR003800	49	50	19.5	12.0	13.2	19.3	20.9	10.8	4.1
MDA093	AMR003810	58	59	0.9	7.0	15.3	22.1	37.4	14.0	2.8
MDA095	AMR003827	11	12	19.4	19.7	16.4	13.9	11.8	12.7	5.6
MDA095	AMR003852	35	36	20.5	15.6	14.0	11.3	10.5	19.6	7.7
MDA095	AMR003868	50	51	26.6	16.6	15.2	12.0	10.4	12.2	3.8
MDA095	AMR003880	62	63	11.0	18.0	18.1	16.6	18.9	13.7	3.6

Figure 13-5
I-Dump Sieve Analysis Results



13.6 Crushability

As the PSD of K-dump is less than the required 10 particle of -75mm/+50mm it was omitted from the crushability testing performed by Coremet. Samples of G-dump and C-dump were analysed to determine SAG Mill Comminution (SMC) and Bond Crushing Work (CWi) parameters. Analysis of the results indicated an average CWi of 10±3.4, with a maximum of 17.7.

10 kg samples of each dump were crushed to -5mm, -3mm, and -1.2mm, and screened at 0.5mm. All samples were sent for HLS testing at 2.9 t/m3 to determine the crushing size to use for the overall process. Results are presented in Table 13-7.

Analysis of these results shows little difference in the mass yield of the coarse fraction between a crush size of 5 mm or 3 mm. To reduce power consumption, a 5 mm crush size was selected.

Table 13-7
HLS Yields of Crushed Samples

C- Dump - HLS@2.9

5mm

Description	Mass (g)	Mass (%)
0.5x5mm	1774.00	64.42
0x0.5mm	980.00	35.58
Total	**2754.00**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	34.95	4.94	3.18
Floats	672.43	95.06	61.23
Total	**707.38**	**100.00**	**64.42**

3mm

Description	Mass (g)	Mass (%)
0.5x3mm	1773.60	63.17
0x0.5mm	1034.00	36.83
Total	**2807.60**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	38.35	5.52	3.49
Floats	656.42	94.48	59.68
Total	**694.77**	**100.00**	**63.17**

1.2mm

Description	Mass (g)	Mass (%)
0.5x1.2mm	1043.30	37.15
0x0.5mm	1765.00	62.85
Total	**2808.30**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	35.41	5.67	2.10
Floats	589.56	94.33	35.05
Total	**624.97**	**100.00**	**37.15**

G- Dump - HLS@2.9

5mm

Description	Mass (g)	Mass (%)
0.5x5mm	2026.20	79.42
0x0.5mm	525.20	20.58
Total	**2551.40**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	61.98	10.01	7.95
Floats	557.43	89.99	71.47
Total	**619.41**	**100.00**	**79.42**

3mm

Description	Mass (g)	Mass (%)
0.5x3mm	1884.70	75.25
0x0.5mm	620.00	24.75
Total	**2504.70**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	82.96	11.08	8.34
Floats	665.87	88.92	66.91
Total	**748.83**	**100.00**	**75.25**

1.2mm

Description	Mass (g)	Mass (%)
0.5x1.2mm	1090.00	40.52
0x0.5mm	1600.00	59.48
Total	**2690.00**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	84.19	12.80	5.19
Floats	573.55	87.20	35.33
Total	**657.74**	**100.00**	**40.52**

K- Dump - HLS@2.9

5mm

Description	Mass (g)	Mass (%)
0.5x5mm	1218.00	51.31
0x0.5mm	1156.00	48.69
Total	**2374.00**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	101.06	13.87	7.11
Floats	627.71	86.13	44.19
Total	**728.77**	**100.00**	**51.31**

3mm

Description	Mass (g)	Mass (%)
0.5x3mm	1255.80	51.07
0x0.5mm	1203.00	48.93
Total	**2458.80**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	105.39	13.97	7.13
Floats	649.22	86.03	43.94
Total	**754.61**	**100.00**	**51.07**

1.2mm

Description	Mass (g)	Mass (%)
0.5x1.2mm	1054.70	43.37
0x0.5mm	1377.00	56.63
Total	**2431.70**	**100.00**

Description	Mass (g)	% Yield	Overall Mass yield
Sink	95.45	15.09	6.54
Floats	537.13	84.91	36.83
Total	**632.58**	**100.00**	**43.37**

13.7 Sepro Dense Media Separation

Sepro Laboratories (Sepro) of Vancouver, Canada performed a two stage DMS pilot plant test on material from G-dump, K-dump, and a K/G blend (84%/16%) to simulate commercial plant feed in 2022. Samples were screened to -5mm/+500µm and processed through a two stage DMS pilot plant. The -500µm material was not used in the tests. A primary cut point of 2.74 t/m^3 and secondary cut point of 2.93 t/m^3 were selected, as this have been shown to produce 6.0wt% Li_2O concentrate for similar materials. The results are summarized in Table 13-8.

Table 13-8
Sepro Pilot DMS Results

G Dump

Description	Weight		Li2O Grade	Li Distribution
	(kg)	(%)	Li2O (%)	Li2O (%)
Sinks (D50 = 2.95)	5.14	2.8	6.00	25.4
Middlings	13.25	7.3	2.83	30.9
Floats (D50 = 2.75)	140.32	77.2	0.30	34.8
Fines (-0.5 mm)	*23.11*	*12.7*	*0.47*	*9.0*

K Dump

Description	Weight		Li2O Grade	Li Distribution
	(kg)	(%)	Li2O (%)	Li2O (%)
Sinks (D50 = 2.95)	3.97	2.8	6.16	17.4
Middlings	11.95	8.5	3.21	27.3
Floats (D50 = 2.74)	65.77	46.5	0.38	17.7
Fines (-0.5 mm)	*59.72*	*42.2*	*0.88*	*37.6*

K/G Blend

Description	Weight		Li2O Grade	Li Distribution
	(kg)	(%)	Li2O (%)	Li2O (%)
Sinks (D50 = 2.95)	4.64	2.7	6.23	17.2
Middlings	14.18	8.3	3.61	30.4
Floats (D50 = 2.74)	89.36	52.3	0.37	19.5
Fines (-0.5 mm)	*62.61*	*36.7*	*0.88*	*32.9*

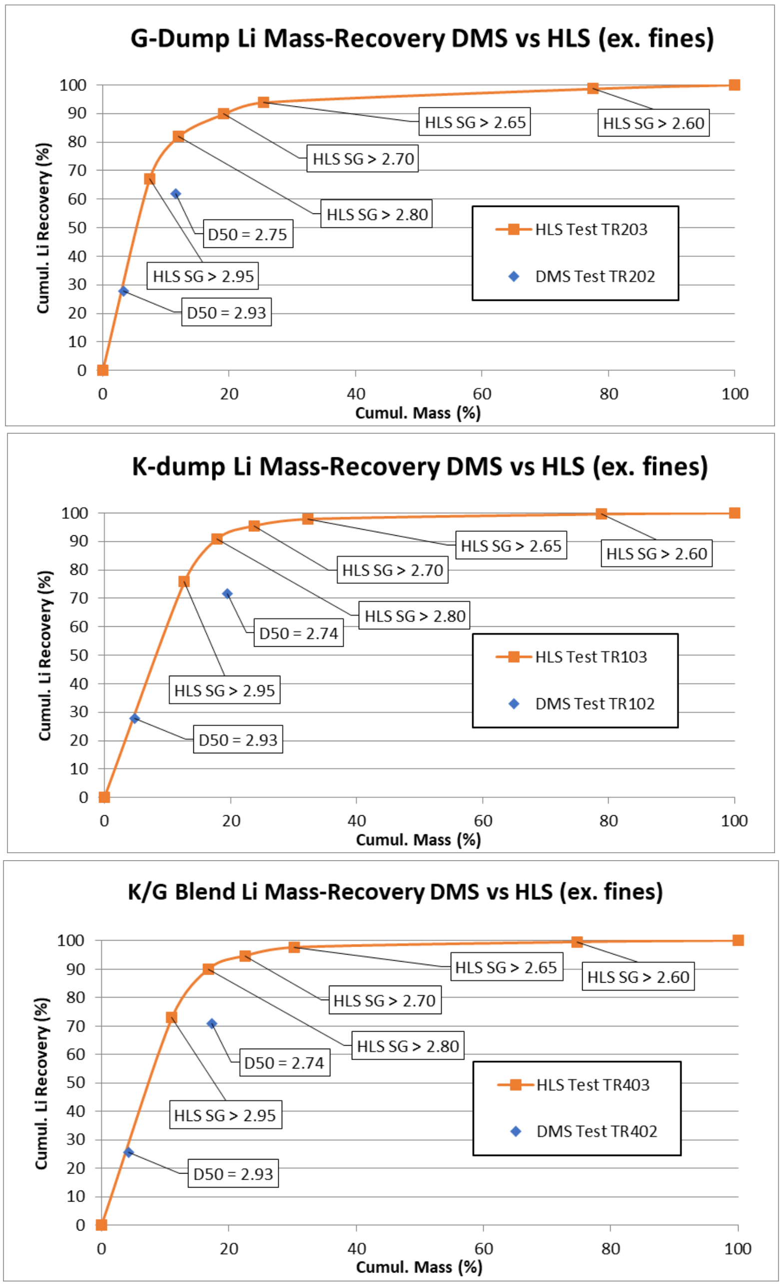
These results indicated that a primary cut point of 2.74 t/m3 and a secondary cut of 2.93 t/m3 can produce a concentrate grade that is above or equal to 6wt% Li2O. The tailings generated by this combination are above the cut-off grade of 0.2wt% Li2O.

Additional HLS tests were conducted at lower t/m3 values to determine the optimal primary cut point to produce tailings below the cut-off grade of 0.2wt% Li2O. Lithium mass recovery curves from the DMS and HLS results were generated and are presented in Figure 13-6.

These HLS test results and mass recovery curves indicated that a primary cut point of near 2.65 t/m3 would produce tailings below the cut-off grade of 0.2wt% Li2O and a final product near 6.0wt% Li2O.

Figure 13-6

Sepro Pilot Li Mass Recovery Curves



13.8 Pesco Dense Media Separation

Pesco Services (Pesco) in Durban, South Africa was requested to run a DMS pilot plant test to match that of Sepro. Pesco selected a primary cut point of 2.75 t/m3 and a secondary cut point of 2.95 t/m3. These results are presented in Table 13-9 and showed that a primary cut point of 2.75 t/m3 would produce tailings above the 0.2wt% Li2O cut off grade and a concentrate grade higher than 6wt% Li2O.

Table 13-9
Pesco Pilot Plant Results

	Primary @ 2.75 cut density					Secondary @ 2.95 cut density					
	Mass (%)	% Li_2O	% Li_2O Recovery	% Fe_2O_3	Fe_2O_3 Recovery %	Mass (%)	% Li_2O	% Li_2O Recovery	% Li_2O Overall Recovery	% Fe_2O_3	Fe_2O_3 Recovery %
Feed		1.10		0.74			4.10			1.90	
Sinks (%)	16.7	4.07	64.05	1.88	85.91	33.8	6.93	57.21	36.64	1.86	69.86
Floats (%)	83.3	0.46	35.95	0.55	14.09	66.2	2.65	42.79	15.38	1.91	30.14
Total	**100**	**1.06**	**100**	**1.40**	**100**	**100**	**4.10**	**100**		**1.88**	**100**

DMS testing is ongoing by Pesco Services (Pesco) in Durban, South Africa in 2023, investigating various cut points for both primary and secondary stages of DMS. To date, primary cut points of 2.55 t/m3, 2.65 t/m3, 2.70 t/m3, and 2.75 t/m3 have been tested and the results are presented in Table 13-10. These results confirm that a primary DMS cut point of 2.65 t/m3 will produce a tailing grade of 0.19wt% Li2O for K-dump, 0.11wt% Li2O for G-dump, and 0.10wt% Li2O for C-dump, all below the 0.2wt% Li2O cut-off grade. It should be noted that the concentrate grade of C-dump produced at the 2.65 t/m3 is only 0.57wt% Li2O and is too low to produce a secondary concentrate grade close to the desired 6wt% Li2O. For this reason, no further testing was investigated on C-dump.

Secondary DMS testing at 2.85 t/m3, 2.90 t/m3, and 2.95 t/m3 is planned on the sink's product of 2.55 t/m3, 2.65 t/m3, and 2.70 t/m3 primary DMS. In the absence of secondary DMS results, the Sepro and Pesco pilot plant results have been interpolated. A secondary DMS cut point of 2.85 t/m3 has been selected for the process design.

Table 13-10
Pesco Primary DMS Results

C-Dump

2.55

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	94.4	58.4	0.45	1.73	82.7	58.6
Floats	67.2	41.6	0.13	1.72	17.3	41.4
Total	**161.6**	**100.0**	**0.32**	**1.72**	**100**	**100**

2.65

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	73.6	46.7	0.57	1.63	83.3	47.6
Floats	84.1	53.3	0.10	1.57	16.7	52.4
Total	**157.7**	**100.0**	**0.32**	**1.60**	**100**	**100**

2.7

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	25.8	16.3	1.03	2.21	53.0	23.0
Floats	133.0	83.7	0.18	1.43	47.0	77.0
Total	**158.8**	**100.0**	**0.31**	**1.56**	**100**	**100**

2.75

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	14.1	8.8	2.44	3.93	67.9	20.5
Floats	145.2	91.2	0.11	1.48	32.1	79.5
Total	**159.3**	**100.0**	**0.32**	**1.69**	**100**	**100**

G-Dump

2.55

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	143.0	83.6	0.81	2.16	97.4	93.7
Floats	28.1	16.4	0.11	0.74	2.6	6.3
Total	**171.1**	**100.0**	**0.69**	**1.93**	**100**	**100**

2.65

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	48.7	28.5	2.14	5.12	88.2	80.1
Floats	122.1	71.5	0.11	0.51	11.8	19.9
Total	**170.7**	**100.0**	**0.69**	**1.82**	**100**	**100**

2.7

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	36.6	20.8	2.85	7.23	86.6	74.7
Floats	138.9	79.2	0.12	0.64	13.4	25.3
Total	**175.5**	**100.0**	**0.69**	**2.02**	**100**	**100**

2.75

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	24.8	14.1	3.99	8.54	80.7	63.9
Floats	150.7	85.9	0.16	0.80	19.3	36.1
Total	**175.5**	**100.0**	**0.70**	**1.89**	**100**	**100**

K-Dump

2.55

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	64.0	68.2	1.67	0.35	95.0	79.5
Floats	29.9	31.8	0.19	0.19	5.0	20.5
Total	**94.0**	**100.0**	**1.20**	**0.30**	**100**	**100**

2.65

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	21.3	22.2	4.32	0.59	86.6	43.6
Floats	74.7	77.8	0.19	0.22	13.4	56.4
Total	**95.9**	**100.0**	**1.11**	**0.30**	**100**	**100**

2.7

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	22.0	23.0	4.15	0.59	86.1	43.4
Floats	73.7	77.0	0.20	0.23	13.9	56.6
Total	**95.7**	**100.0**	**1.11**	**0.31**	**100**	**100**

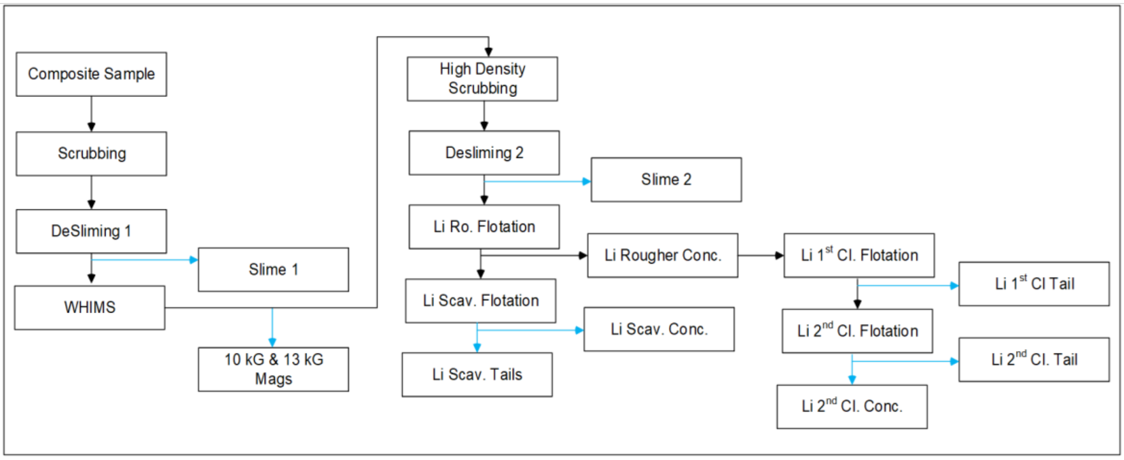
2.75

Description	Mass (kg)	% Yield	% Li_2O	% Fe_2O_3	% Li_2O Recovery	Fe_2O_3 Recovery %
Sinks	13.0	14.2	5.23	0.54	70.2	25.4
Floats	78.5	85.8	0.37	0.26	29.8	74.6
Total	**91.5**	**100.0**	**1.05**	**0.30**	**100**	**100**

13.9 Flotation Testing

SGS Canada Inc. (SGS) of Lakefield, Canada conducted flotation testing, in 2023 on K-dump and G-dump samples passing 500µm (fresh feed) generated by Sepro, as well as middlings (secondary DMS floats) produced from Sepro's DMS pilot testing. Samples were ground to a P100 of 300µm and passed through a Knelson concentrator for heavy mineral removal. Knelson tailings were sent through magnetic separation prior to being processed in a two-stage mica reverse flotation, followed by a three stage spodumene flotation. The complete testing flowsheet is shown in Figure 13-7.

Figure 13-7
SGS Flotation Test Flowsheet



Results on the fresh feed samples of K-dump show that a final concentrate of 5.9wt% Li2O can be produced. The concentrate generated from the fresh feed of G-dump is 2.65wt% Li2O.

Additional tests were performed on a sample that blended the middlings from the K-dump with the K-dump fresh feed. The concentrate produced was 6.74wt% Li2O and these results are presented in Table 13 11. G-dump middlings were processed without blending in fresh G-dump feed and results showed that a 6.44wt% Li2O concentrate can be produced, shown in Table 13-11.

Additional testing on a composite sample blending fresh feed and middling of both K-dump and G-dump is planned.

Table 13-11

Flotation Results on K dump fresh feed and middlings

Test No. / Objective	Product	Weight g	Weight %	Assays % Li	Assays % Li_2O	Assays % K_2O	Assays % Fe_2O_3	Distribution % Li	Distribution % K_2O	Distribution % Fe_2O_3
F4	F4 Li 2nd Cl Conc.	223	10.4	3.13	6.74	0.28	0.82	56.1	1.1	5.9
K Dump	F4 Li 1st Cl Conc.	274	12.7	2.97	6.40	0.39	0.82	65.4	1.9	7.3
Midd& Fines	F4 Li Ro. Conc.	338	15.7	2.65	5.70	0.64	0.80	71.8	3.9	8.7
-300 mic	F4 Li Ro & Scav Conc	391	18.1	2.41	5.19	0.81	0.80	75.7	5.7	10.1
Based on F2 but on the DMS U/S + Middling	F4 Li Ro Tail	1284	59.6	0.06	0.14	2.58	0.34	6.6	60.4	14.1
	F4 Li Ro Scav Tail	1232	57.2	0.03	0.06	2.61	0.32	2.8	58.6	12.7
	F4 Mica Conc.	79.1	3.7	0.39	0.85	7.96	2.31	2.5	11.5	5.9
	F4 Mag Conc	62.3	2.9	0.56	1.21	3.31	9.77	2.8	3.8	19.6
	F4 Knelson Conc.	68.1	3.2	0.42	0.90	3.43	2.48	6.3	2.3	3.7
	F4 Total Slimes	322	15.0	0.38	0.83	3.09	4.63	9.9	18.1	48.0
	Head (calc.)	2153	100	0.58	1.24	2.55	1.44	100	100	100
	Head (calc.fines + Mid.)			0.61	1.30	2.23	1.33			

Table 13-12

Flotation Results on G dump middlings

Test No. / Objective	Product	Weight g	Weight %	Assays % Li	Assays % Li_2O	Assays % K_2O	Assays % Fe_2O_3	Distribution % Li	Distribution % K_2O	Distribution % Fe_2O_3
F3	F3 Li 2nd Cl Conc. Non-mag	549	25.8	2.99	6.44	0.18	0.45	63.3	3.2	1.4
G Dump Middlir	F3 Li 2nd Cl Conc.	580	27.3	2.88	6.21	0.19	1.25	64.5	3.6	4.1
-300 mic	F3 Li 1st Cl Conc.	639	30.1	2.78	5.99	0.22	1.26	68.5	4.5	4.5
First Trial for Floating Spodumene from G-Dump Middling Sample	F3 Li Ro. Conc.	750	35.3	2.52	5.43	0.30	1.20	73.0	7.2	5.0
	F3 Li Ro & Scav Conc	797	37.5	2.42	5.21	0.32	1.18	74.4	8.1	5.3
	F3 Li Ro Tail	470	22.1	0.10	0.22	1.20	0.41	1.8	18.0	1.1
	F3 Li Ro Scav Tail	424	19.9	0.03	0.05	1.26	0.36	0.4	17.1	0.9
	F3 Mica Conc.	160	7.5	0.74	1.58	6.11	2.37	4.6	31.4	2.1
	F3 Mag Conc	350	16.4	0.36	0.78	1.47	33.56	4.9	16.4	65.6
	F3 Knelson Conc.	71.6	3.4	1.35	2.91	1.07	12.90	3.7	2.5	5.2
	F3 Total Slimes	324	15.2	0.96	2.08	2.37	11.62	12.0	24.6	21.0
	Head (calc.)	2126	100	1.22	2.62	1.47	8.42	100	100	100
	Head (Dir.)	0	0	1.22	2.63	1.49	8.93			

13.10 Reflux Classifier

Nagrom of Australia is conducting batch Reflux Classifier (RC) tests in 2023 to check for effective mica removal from C-dump, G-dump, and K-dump at various size fractions.

Should these results prove positive, trade off studies should be conducted to confirm if the RC can be added into the process.

13.11 Processing Flowsheet

A process flowsheet was developed for K-dump and G-dump as illustrated in Figure 17 1. C-dump is excluded from the process as it showed very low concentrate grades in during testing. The conceptual flowsheet includes the following:

a) A crushing circuit to crush all material to -5 mm size prior to beneficiation.

b) Splitting the crushed material into a coarse fraction (-5 mm +0.5 mm) and fine fraction (-0.5 mm) to be processed in two separate circuits.

c) Processing the coarse material through a two stage DMS plant. Secondary cyclone overflow (middlings) are sent to the grinding plant.

d) Processing the -0.5 mm material will be processed through a grinding and flotation plant.

14 MINERAL RESOURCE ESTIMATES

On behalf of Tantalex, MSA completed a Mineral Resource estimate for the Manono Tailings deposits.

To the best of the QP'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 Resources presented herein, with an effective date of 23 August 2023, represent an update to the previous Manono tailings deposits dated 13 December 2022. The updated estimates incorporate drillhole data completed by Tantalex from September 2021 to July 2022 and, in the QP's opinion, were collected using reasonable procedures and protocols.

The Mineral Resource was estimated using the 2019 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 estimates were conducted using Datamine Studio RM software, together with Leapfrog Geo, which was used for the modelling of three-dimensional volumes. Microsoft Excel, JMP statistical software and Datamine Supervisor were used for data analysis.

14.1 Mineral Resource Estimation Database

The principal sources of information used for the estimate are the exploration drilling conducted by Tantalex from September 2021 to July 2022. The database provided by Tantalex to inform the Mineral Resource estimates consists of:

- Information from diamond drillholes in the form of:
    - Collar surveys.
    - Downhole Surveys – all holes were vertically drilled and were not surveyed.
    - Sampling and assay data.
    - Geology data.
- Specific gravity (SG) measurements from pits excavated to one metre below the surface of the tailings.
- Topographic surveys provided as contours in GIS shapefile format.

The drillhole and SG data were provided as Microsoft Excel files.

A total of 367 drillholes, amounting to 11 962 metres of drilling, were completed across nine tailings deposits. The number of drillholes and metres drilled per deposit is summarised in Table 14-1.

Table 14-1 Number of drillholes and total metres drilled per deposit		
Deposit	**Number of Drillholes**	**Metres Drilled**
Cc	34	2 312
Cf	4	136
Ec	32	1 854
Gc	24	1 479
Gf	50	886
Hc	21	1 260
Hf	26	689
Ic	20	1 226
K	156	2 120
Total	**367**	**11 962**

14.2 Exploratory Data Analysis of the Raw Data

The dataset examined consisted of sampling and logging data from aircore drillholes. The following attributes are of direct relevance to the estimate:

- Lithium (Li), tin (Sn) and tantalum (Ta) in parts per million (ppm).
- Specific gravity measurements.
- Lithological logs.

Lithium grades in parts per million were converted to percentage lithium oxide (Li_2O) by applying a factor of 2.153 and then converting ppm to percent.

A total of 8 038 metres of drillhole samples were assayed, however not all samples were assayed for all three elements. A summary of assayed metres is shown in Table 14-2.

Table 14-2 Assayed metres per deposit		
Deposit	**Drilled Metres**	**Assayed Metres**
Cc	2 312	860
Cf	136	135
Ec	1 854	661
Gc	1 479	1 453
Gf	886	866
Hc	1 260	600
Hf	689.4	432
Ic	1 226	974
K	2 120	2 057

Due to insufficient data coverage, a Mineral Resource estimate was not completed for the Cf deposit.

14.2.1 Validation of the data

MSA undertook a high-level validation process which included the following checks:

- Examining the sample assay, collar survey and geology data to ensure that the data are complete for all the drillholes,
- Examining the de-surveyed data in three dimensions to check for spatial errors,
- Examination of the assay and density data to ascertain whether they are within expected ranges,
- Check for “FROM-TO” errors, to ensure that the sample data do not overlap one another or that there are no unexplained gaps in the sampling.

The data validation exercise revealed the following:

- There are no unresolved errors relating to missing intervals and no overlaps in the drillhole logging data. Absent assays correspond to intervals where no samples were taken or unassayed values.
- Examination of the drillhole data in three dimensions shows that the collars of the drillholes surveyed by DGPS plot generally in their expected positions relative to the topographic surface. Where noticeable deviations were noted, Tantalex provided updated topographic data which corrected any issues identified.
- Extreme assays were checked, and no errors were found.
- No assays were returned for four samples - AMR5323, AMR5326, AMR5331 and AMR5432.

- Seven samples returned values at the upper limit of detection (10 000 ppm) with no over limit analysis undertaken. These were found to only affect CRM samples and this issue does not impact the Mineral Resource.
- Nine samples reported tin grades at the upper limit of detection(10 000 ppm) with no over limit analysis undertaken. The tin values for these samples were set to the upper detection limit value of 10 000 ppm.
- Samples that reported below detection limit values were set to half the detection limit value.

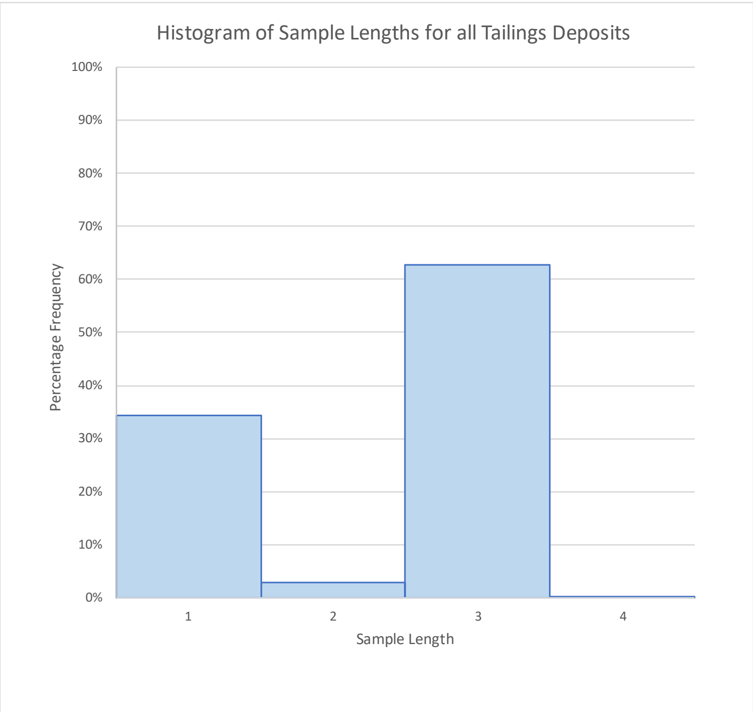
14.2.2 Statistics of the Raw Sample Data

14.2.2.1 Sample Lengths

Samples were taken at 1 metre intervals during the early phase of exploration, however, the sampling methodology was later changed to three metre composites. As a result, 34% of the total samples were taken at one metre intervals including the Cc, Gc, Hc, Hf and Ic deposits. The remainder of the samples were mostly taken at three metre intervals.

A histogram of the sample intervals for the combined nine deposits is shown in Figure 14-1. Samples taken at 2 m and 4 m make a small percentage of the samples, which tend to occur along the base of the deposits and are not representative of the total drilled tailings.

Figure 14-1
Sample Length Histogram for Manono Samples



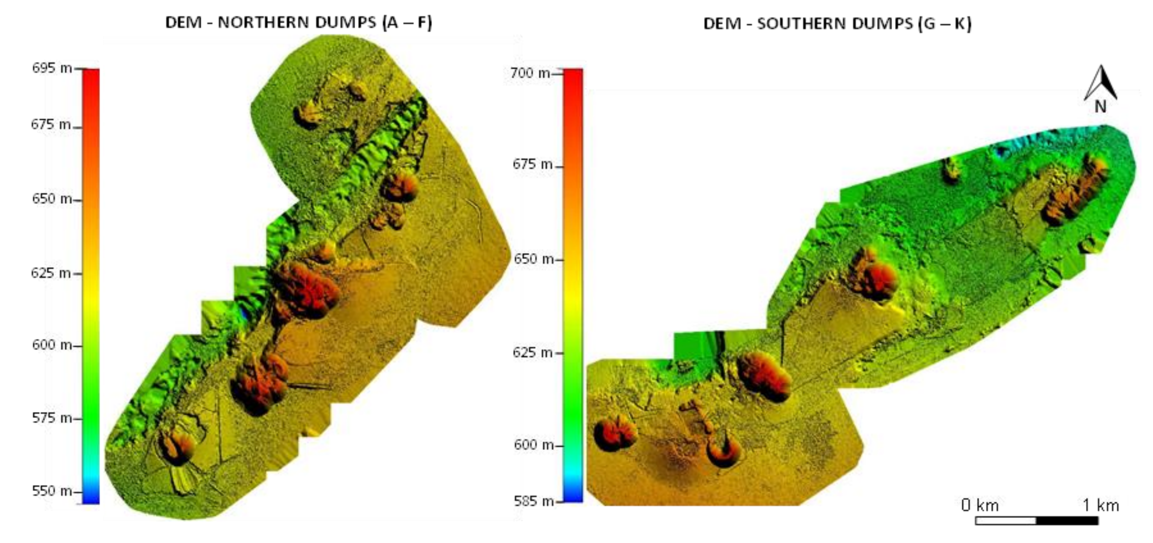
14.3 Geological Modelling

14.3.1 Topography

A topographic digital surface model (DSM) covering the Manono Lithium Tailings Project area was provided by Tantalex. An unmanned aerial vehicle (UAV), photogrammetry topographical survey and volumetric estimation of the Project was conducted by Ikigai Environmental Specialists during September 2022. The survey was conducted within UTM Zone 35 S and referenced to the WGS84 datum.

The survey area spanned a total of 1,309.4 hectares and the data was processed using Pix4D software to provide 1 m contour interval digital elevation models (DEMs) (Ikigai, 2021) (Figure 4-1). Geovia Surpac was used to calculate the final volumetric estimates (Ikigai, 2021).

Figure 14-2
Manono Lithium Tailings Project DEMs



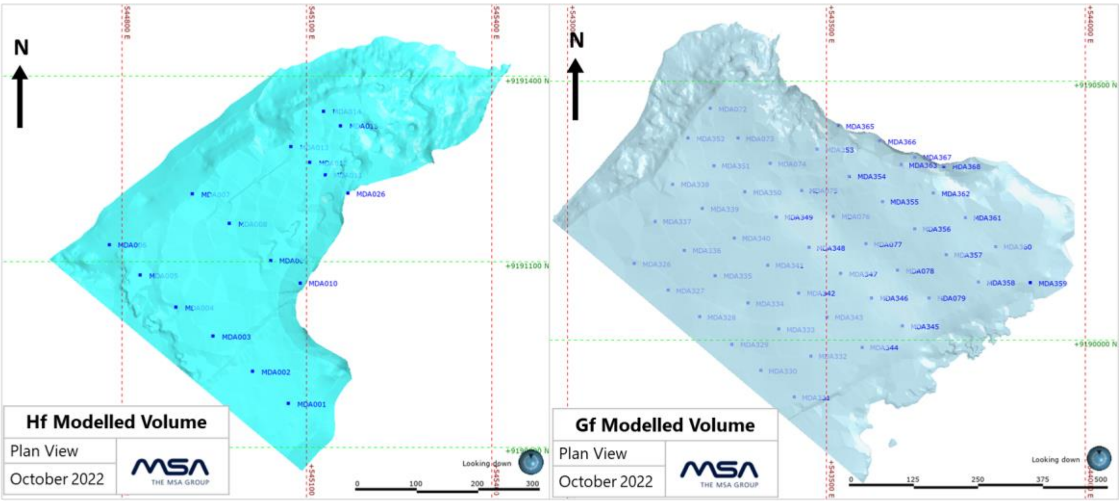
Source: *Adapted from Ikigai (2021)*

14.3.2 Tailings Volumes

Leapfrog Geo was used to generate three-dimensional volumes representing the tailings deposits. The upper limit of the tailings deposits was defined using the supplied topographical surveys. Due to the absence of a pre-depositional surface, the base of a deposit was interpreted to occur where laterite material or saprolite was intercepted. Since many of Manono's deposits consist of large volumes of laterite, the base was interpreted to occur where the last laterite horizon was intercepted in each drillhole. In the absence of a basal laterite, grade data was used to guide the modelling.

The volume of each deposit was generated by intercepting the modelled base with the topography. The exception to this being the Hf and Gf deposits, which remain unexplored in the southwest, therefore the extent of the volumes was limited to half the drillhole spacing in this direction (Figure 14-3).

Figure 14-3
Volumes for the Hf and Gf deposits



As the lithology logging is recorded on one metre intervals while the majority of the samples were assayed at three metre intervals, there were instances where basal laterite was found to contain significant lithium grades due to sample compositing taking place across lithology types. This was found to particularly impact the K tailings, therefore a combination of lithology and grade data was used to define the base of the deposit.

Several of the deposits consist of a combination of material types, including laterite, pegmatite and clay. Where sufficient data was available, volumes for each material type were modelled, with each deposit being treated as a separate domain. In the absence of data, an angle of repose between 30° and 35° was assumed when modelling each layer.

Volumes for the Ic and Hc deposits are presented in Figure 14-4 and Figure 14-5.

Figure 14-4
Modelled volumes of the Ic Tailings Deposit

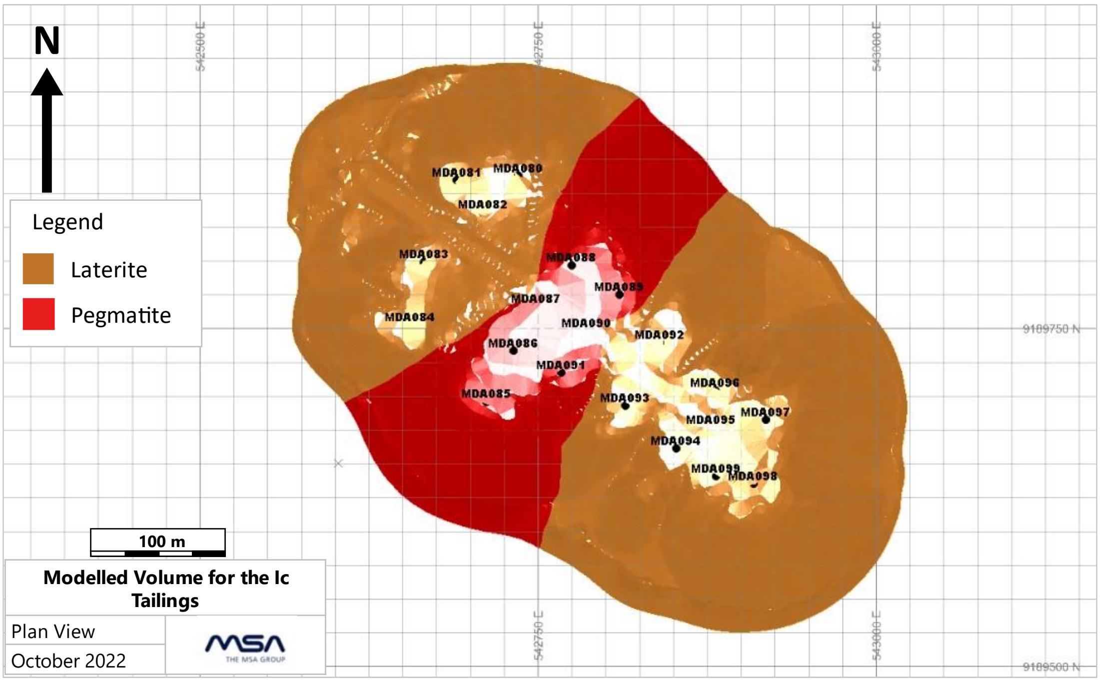
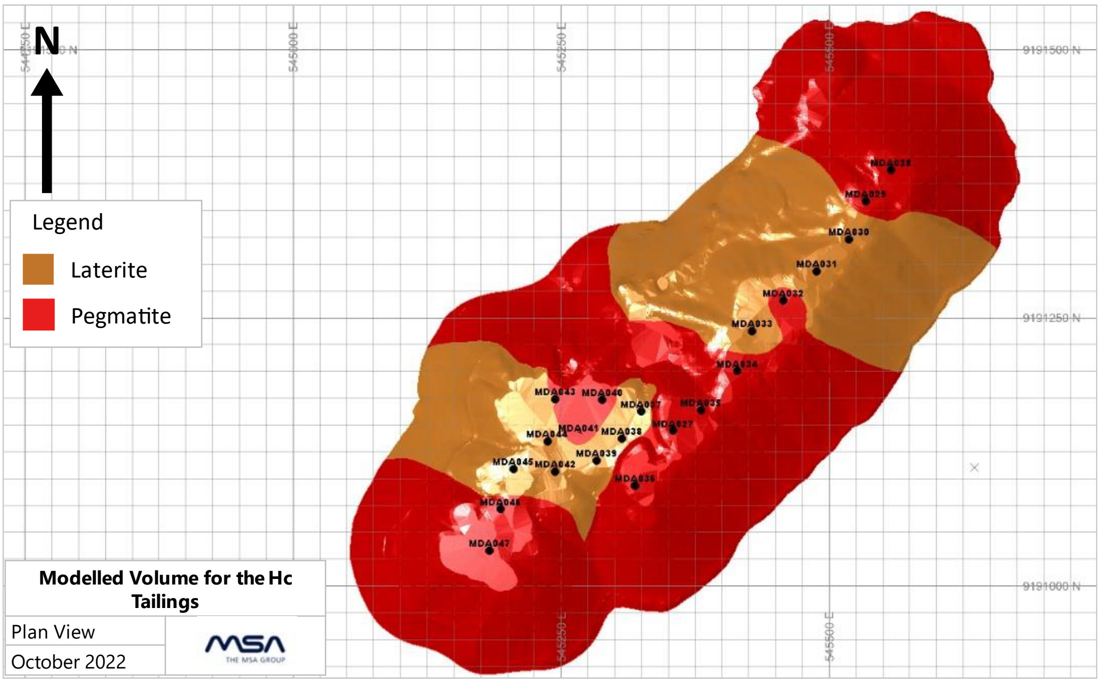
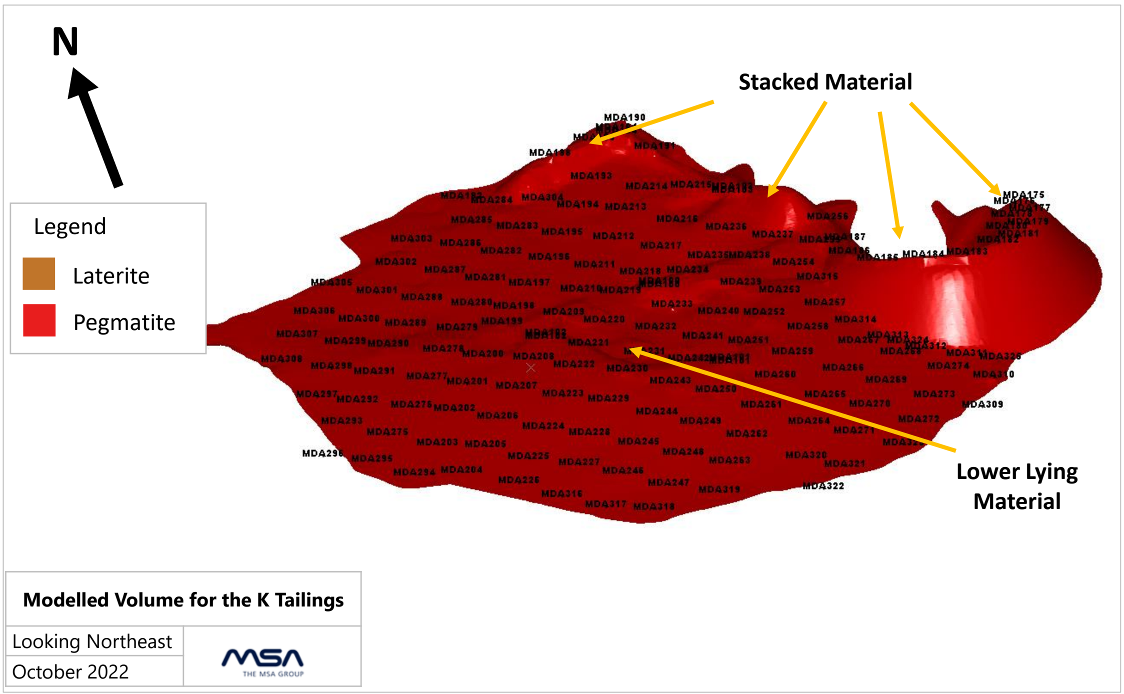


Figure 14-5
Modelled volumes of the Hc Tailings Deposit



The K deposit is exclusively composed of pegmatite material. Figure 14-6 shows the modelled volume, with the stacked material visible in the background while the thin, lower lying material is shown in the foreground. During estimation, the stacked material was separated from the lower lying tailings using a digitised polyline.

Figure 14-6
Modelled volumes of the K Tailings Deposit



The modelled lithological zones for each deposit were treated as discrete estimation domains, therefore, each volume was given an identifier number. A summary of the volumes modelled for each deposit is presented in Table 14-3.

Table 14-3
Number of volumes per material type modelled for each deposit

Material Type	Cc	Ec	Gc	Gf	Hc	Hf	Ic	K
Pegmatite	1	3	3	2	3	1	2	1
Laterite	0	2	3	1	2	1	2	0
Clay	0	0	0	2	0	0	0	0

14.4 Statistical Analysis of the Composite Data

Samples were composited to 3 m lengths using length weighting.

14.4.1 Lithium Oxide (Li_2O)

Summary statistics for lithium oxide for the three metre composite samples are presented in Table 14-4.

The highest Li_2O grades are present in the K dump with the stacked material having a mean Li_2O grade of 0.66%, while the lower lying material has a mean grade of 0.85% Li_2O. The grade variability is typically low for all the domains, as seen by the low coefficient of variation (CV) values. This is with the exception of the Cc and Gc dumps which have CV values larger than 1. Higher lithium grades typically occur in pegmatite tailings, with laterites generally reporting lithium grades below 0.10% Li_2O.

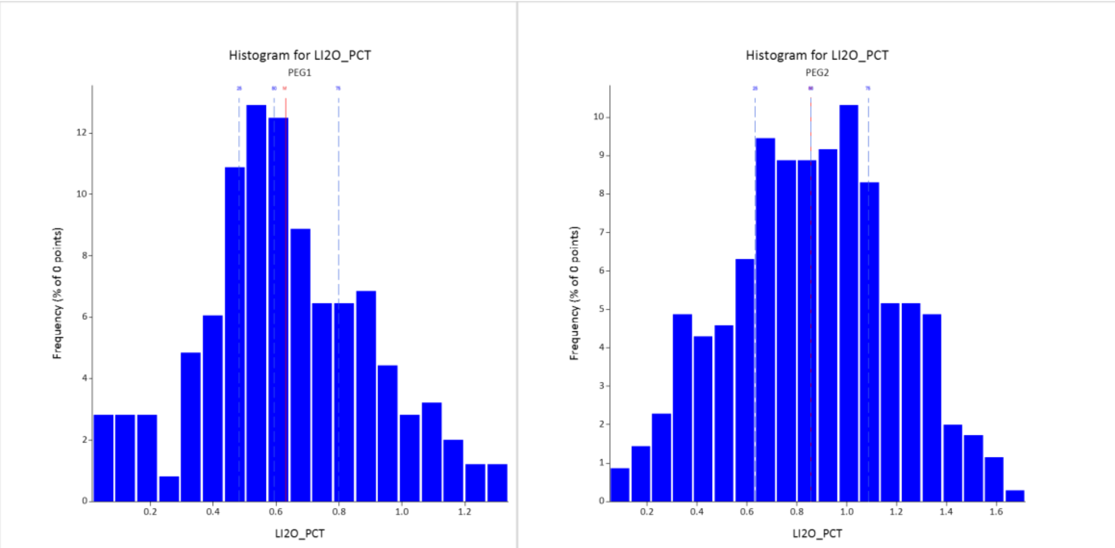
Table 14-4
Summary statistics for lithium oxide per domain

Domain	Number of Composites	Minimum %	Maximum %	Mean %	CV
Cc Dump					
PEG1	271	0.02	0.98	0.14	1.16
Ec Dump					
LAT1	38	0.03	0.27	0.06	0.57
LAT2	30	0.03	0.09	0.05	0.27
PEG1	28	0.01	0.13	0.05	0.53
PEG2	87	0.03	0.21	0.07	0.40
PEG3	21	0.04	0.10	0.06	0.23
Gc Dump					
LAT1	19	0.02	0.09	0.04	0.46
LAT2	338	0.01	0.39	0.04	0.77
LAT3	2	0.02	0.05	0.04	0.51
PEG1	15	0.02	0.37	0.08	1.17
PEG2	85	0.02	1.41	0.31	1.29
PEG3	4	0.05	0.08	0.06	0.18
Gf Dump					
CLA1	59	0.02	0.56	0.16	0.42
CLA2	16	0.07	0.17	0.12	0.27
LAT1	70	0.02	0.22	0.10	0.44
PEG1	81	0.00	0.72	0.24	0.81
PEG2	7	0.01	0.09	0.03	0.82
Hc Dump					
LAT1	49	0.02	0.07	0.03	0.35
LAT2	27	0.01	0.08	0.03	0.55
PEG1	10	0.03	0.06	0.04	0.32
PEG2	93	0.02	0.28	0.08	0.55

Domain	Number of Composites	Minimum %	Maximum %	Mean %	CV
PEG3	13	0.03	0.07	0.04	0.18
Hf Dump					
LAT1	28	0.01	0.14	0.04	0.70
PEG1	91	0.01	0.18	0.09	0.37
Ic Dump					
LAT1	140	0.01	0.18	0.04	0.60
LAT2	72	0.00	0.15	0.04	0.70
PEG1	77	0.01	0.26	0.09	0.63
PEG2	19	0.04	1.05	0.39	0.69
K Dump					
PEG1	237	0.11	1.34	0.66	0.38
PEG2	356	0.05	1.72	0.85	0.39

Histograms for Li_2O grade for the two domains of the K dump are shown in Figure 14-7. The two distributions approximately resemble the bell curve of a normal distribution, particularly PEG2, while PEG1 shows a slight positive skewness.

Figure 14-7
Sample histograms for Li_2O for the stacked tailings (PEG1) and the lower lying tailings (PEG2)



14.4.2 Tin

Summary Statistics for tin for the three metre composite samples are presented in Table 14-5.

For the five dumps that were estimated, there is very little variability in mean tin grades between the dumps. This lack of variability extends between the material types, where there is very little difference in mean tin grades between pegmatite and laterite material. An exception to this being

PEG 2 in the Gc dump and the two pegmatite layers of the Ic dump. For the K dump, the stacked tailings (PEG1) have a mean tin grade that is almost double the lower lying, fine material (PEG1).

Table 14-5
Summary statistics for tin per domain

Domain	Number of Composites	Minimum ppm	Maximum ppm	Mean ppm	CV
Gc Dump					
LAT1	19	139	587	245	0.47
LAT2	328	26	1693	250	0.73
LAT3	2	114	415	265	0.57
PEG1	18	31	655	213	0.83
PEG2	83	50	5296	470	1.75
PEG3	4	213	267	232	0.09
Gf Dump					
CLA1	59	60	308	143	0.28
CLA2	16	93	322	198	0.27
LAT1	70	35	315	146	0.36
PEG1	81	68	789	190	0.60
PEG2	7	62	285	197	0.37
Ic Dump					
LAT1	140	124	1640	370	0.61
LAT2	129	94	940	361	0.55
PEG1	96	136	1680	573	0.50
PEG2	19	205	863	491	0.40
K Dump					
PEG1	226	27	2265	662	0.45
PEG2	352	5	4613	319	0.91

14.4.3 Tantalum

Summary Statistics for tantalum for the three metre composite samples are presented in Table 14-6.

The average tantalum grades across the five estimated tailings do not differ substantially, this lack of variability is observed between the material types as well. The stacked tailings of the K dump (PEG1) report a slightly higher mean Ta grade of 33 ppm, while the low-lying material of the K dump has a mean Ta grade that is within range of the other deposits.

Table 14-6
Summary statistics for tantalum per domain

Domain	Number of Composites	Minimum ppm	Maximum ppm	Mean ppm	CV
Gc Dump					
LAT1	19	8	33	19	0.40
LAT2	328	2	148	21	0.83
LAT3	2	11	16	13	0.18
PEG1	18	6	41	13	0.62
PEG2	83	5	75	23	0.62
PEG3	4	12	35	27	0.33
Gf Dump					
CLA1	59	7	27	17	0.26
CLA2	16	16	47	24	0.29
LAT1	70	5	38	20	0.33
PEG1	81	10	200	25	0.89
PEG2	7	8	20	13	0.31
Ic Dump					
LAT1	140	5	52	14	0.55
LAT2	72	5	173	15	1.12
PEG1	77	7	34	19	0.70
PEG2	19	8	83	28	0.50
K Dump					
PEG1	226	4	149	33	0.45
PEG2	352	0	121	25	0.48

14.5 Cutting and Capping

14.5.1 Lithium Oxide

Histograms and log probability plots for each domain were examined for outliers. A decision to apply capping to a domain was guided by breaks in the distribution of each variable and the spatial location of the outlier samples relative to one another.

The capping typically affected three or less samples per domain (Table 14-7).

Table 14-7
Capping for Li_2O grade per domain for each deposit

Deposit	Domain	Number of Composites	Uncapped Mean %	Uncapped CV	Cap Value %	Number of Composites Capped	Capped Mean %	Capped CV
				Ec				
Ec	PEG1	38	0.06	0.57	0.078	3	0.05	0.37
	PEG2	312	0.07	0.40	0.126	2	0.07	0.35
	LAT1	38	0.06	0.57	0.099	2	0.06	0.32
				Gc				
Gc	PEG1	15	0.37	1.17	0.128	3	0.06	0.71
	LAT2	338	0.04	0.77	0.181	2	0.04	0.65
				Gf				
Gf	CLA1	59	0.16	0.42	0.226	2	0.15	0.26
				Hc				
Hc	PEG3	13	0.08	0.55	0.049	1	0.04	0.11
				Hf				
Hf	LAT1	28	0.04	0.70	0.059	3	0.03	0.47
				Ic				
Ic	LAT1	140	0.04	0.60	0.086	3	0.04	0.46

14.5.2 Tin

Capping of tin outliers impacted only three deposits and six domains in total as shown in Table 14-8. Generally, the capping affected two or three samples, with six samples being capped for the stacked tailings of the K dump.

Table 14-8
Capping for Sn grade per domain for each deposit

Deposit	Domain	Number of Composites	Uncapped Mean ppm	Uncapped CV	Cap Value ppm	Number of Composites Capped	Capped Mean ppm	Capped CV
				Gc				
Gc	LAT2	328	250	0.73	787	2	258	0.57
	PEG2	83	470	1.75	984	3	382	0.69
				Gf				
Gf	CLA1	59	143	0.28	245	2	142	0.25
	PEG1	81	190	0.60	545	2	187	0.54
				K				
K	PEG1	226	662	0.45	1365	6	653	0.40
	PEG2	352	319	0.91	1259	2	310	0.60

14.5.3 Tantalum

The capping of tantalum outliers affected four deposits, representing a total of ten domains (Table 14-9). The capping had a minimal impact on the mean tantalum grades, with the only discernible difference being in the CV values. LAT2 of the Ic dump registered the largest decrease in variability due to capping.

Table 14-9
Capping for Ta grade per domain for each deposit

Deposit	Domain	Number of Composites	Uncapped Mean ppm	Uncapped CV	Cap Value ppm	Number of Composites Capped	Capped Mean ppm	Capped CV
Gc								
Gc	PEG1	18	13	0.62	27	1	13	0.52
	PEG2	83	23	0.62	44.4	4	22	0.51
	LAT2	328	21	0.83	71	6	21	0.63
Gf								
Gf	CLA2	16	24	0.29	29	2	23	0.17
	PEG1	81	25	0.89	68.7	2	24	0.51
Ic								
Ic	LAT1	140	14	0.45	34.5	3	13	0.44
	LAT2	72	15	1.12	27.4	2	14	0.40
	PEG2	8	28	0.50	43.1	1	25	0.41
K								
K	PEG1	226	33	0.45	87.7	3	33	0.41
	PEG2	352	25	0.48	63.3	5	25	0.41

14.6 Geostatistical Analysis

Geostatistical analysis was conducted using Datamine Supervisor software. The grade data were transformed to normal scores for modelling purposes and the sills were back transformed for use in estimation. The large majority of the Manono tailings deposits lack sufficient data coverage to model semivariograms, with the exception of the low-lying material of the K dump, which was drilled on a 40 m by 40 m grid.

Experimental semivariograms were calculated for the normal score transformed 3 m composite data. The nugget effect was determined by extrapolating from the first two experimental points of the down-hole semivariogram. The nugget effect for Li_2O grade is low, which is expected due to the low variability observed in the data while the nugget effect for tin grade and tantalum grade was observed to be higher.

Semivariogram maps for the K dump did not indicate the presence of anisotropy in the grade continuity, therefore isotropic semivariogram models were modelled in the horizontal plane resulting in double structured, spherical models for the three elements.

The semivariogram models for Li_2O are presented in Figure 14-8 and the parameters for all three elements are presented in Table 14-10.

Figure 14-8
Semivariograms for the K dump for Li_2O %

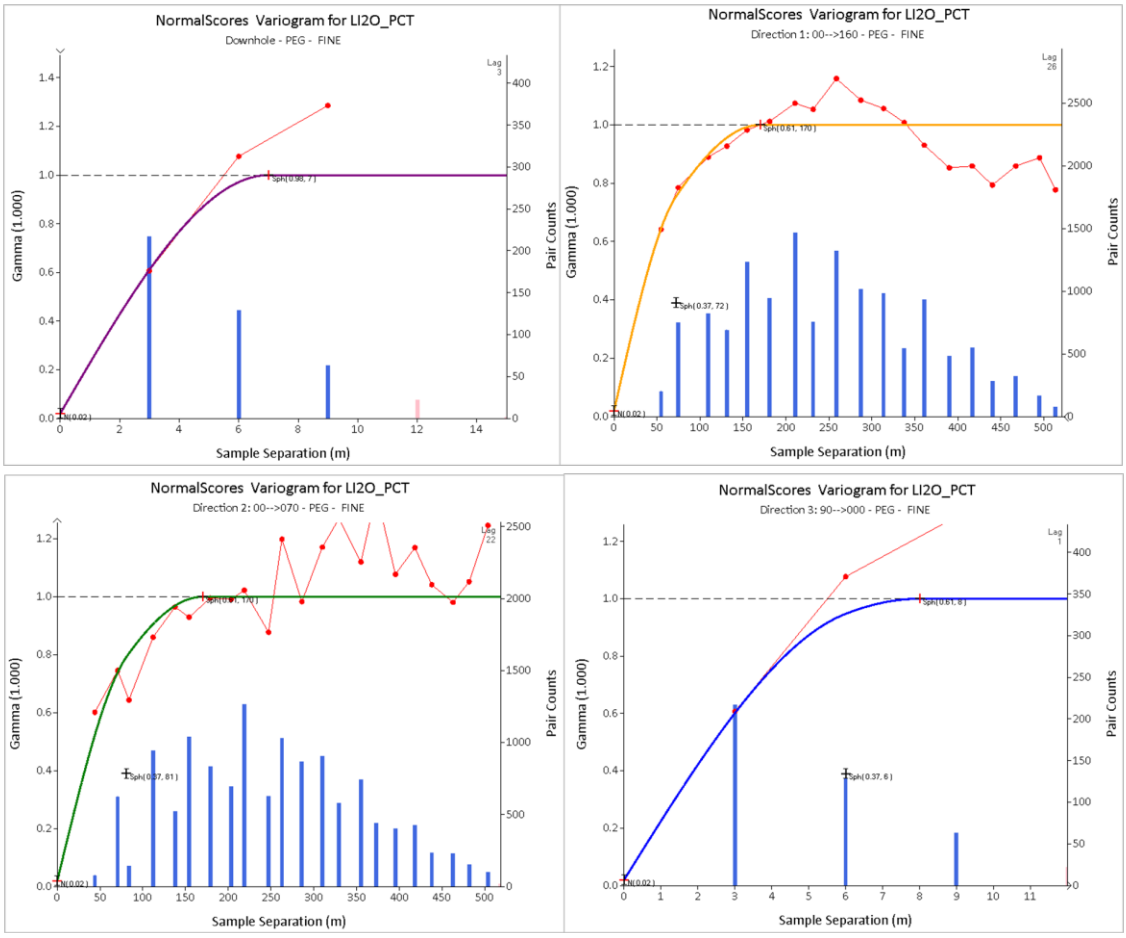


Table 14-10
Semivariogram parameters for K dump

Attribute	Rotation Angles			Rotation Axis			Nugget Effect (C0)	Range (m) of First Structure (R1)			Sill 1 (C1)	Range (m) of Second Structure (R2)			Sill 2 (C2)
	1	2	3	1	2	3		1	2	3		1	2	3	
Li_2O %	0	0	70	Z	X	Z	0.02	72	81	6	0.37	170	170	8	0.61
Sn ppm	0	0	70	Z	X	Z	0.51	100	60	7	0.13	140	140	13	0.36
Ta ppm	0	0	70	Z	X	Z	0.43	57	84	5	0.28	160	160	10	0.29

14.7 Block Modelling

Block models covering each deposit were created using a parent cell of 20 mX by 20 mY by 3 mZ. Sub-celling was applied to optimally fill the modelled volumes, resulting in a minimum sub-cell of 2 mX by 2 mY by 0.5 mZ.

The common origin and block parameters for each deposit are presented in Table 14-11.

Table 14-11
Model prototype origins and block sizes for Manono tailings deposits

Deposit	Model Origin			Block Size			Number of Cells		
	X	Y	Z	X	Y	Z	X	Y	Z
Cc	549900	9195100	600	20	20	3	37	37	40
Ec	549600	9194400	600	20	20	3	30	32	40
Hc	545000	9190900	600	20	20	3	35	35	40
Hf	544650	9190700	600	20	20	3	43	40	35
Ic	542500	9189500	600	20	20	3	30	30	40
Gc	543300	9190250	600	20	20	3	35	35	35
Gf	542950	9189700	600	20	20	3	53	50	35
K	541650	9188850	625	20	20	3	50	40	34

14.8 Estimation Parameters

Attributes were estimated into the modelled volumes using the 3 m composite drillhole sample data by inverse distance squared (IDW2) for all deposits with the exception of the low-lying tailings of the K dump, which was estimated by ordinary kriging (OK). The stacked tailings of the K dump were estimated by IDW2.

The search distance and rotation angles of the OK estimates were based on the semivariogram ranges. Kriging Neighbourhood Analysis (KNA) was used to determine the minimum and maximum number of samples to be included in the search neighbourhood for the OK estimates and the appropriate discretisation points to be used in a parent cell. The KNA exercise considered kriging efficiency and slope of regression values to quantify the level of conditional bias when selecting the optimal parameters.

The estimates were carried out in three passes. The first pass OK estimate applied the variogram ranges, while the second pass expanded the search volume by a factor of 1.5, while the third pass expanded this volume by a factor of 10 to ensure that all blocks received an estimate. A minimum of 5 and a maximum of 10 samples were used in the first two passes, with the third pass estimate allowing a maximum of 12 samples. A limit of two samples per drillhole was imposed on the estimates. Where domains had less than five samples, the mean composite grade was assigned to the blocks.

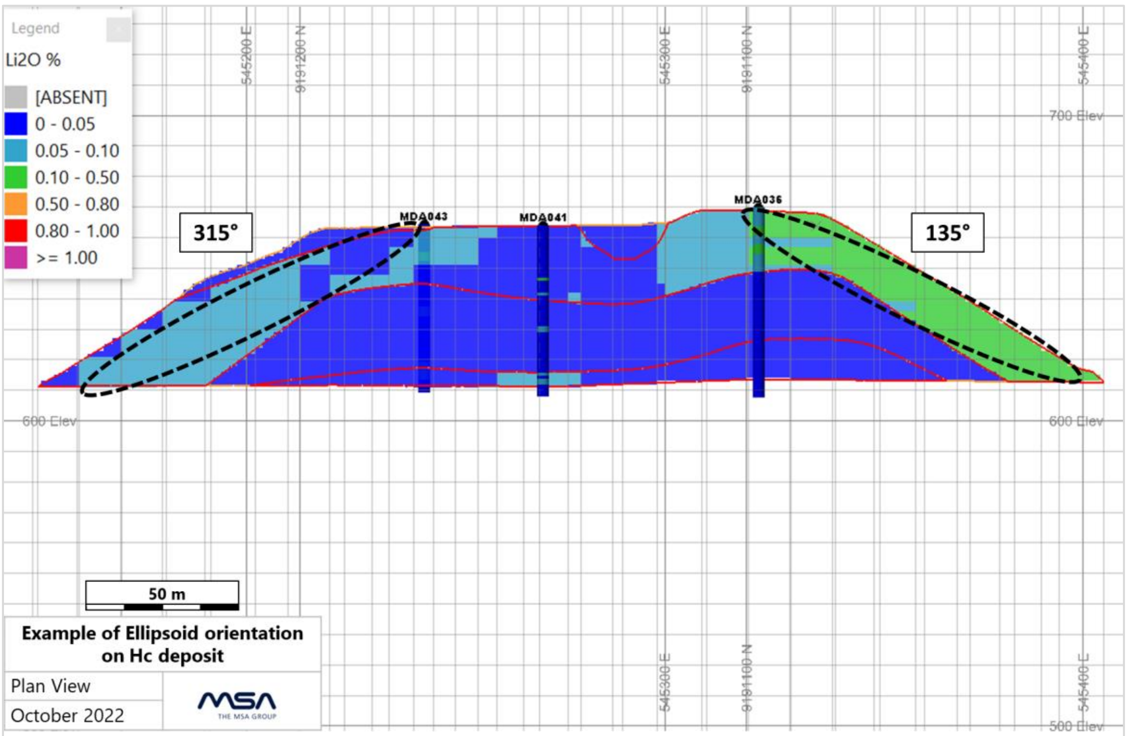
The search parameters for the K dump OK estimates are shown in Table 14-12.

Attribute	Rotation Angles			Rotation Axis			Search Distance (m)			Number of Composites	
	X	Y	Z	X	Y	Z	X	Y	Z	Min	Max
Li_2O %	0	0	70	Z	X	Z	170	170	8	5	10
Sn ppm	0	0	70	Z	X	Z	140	140	13	5	10
Ta ppm	0	0	70	Z	X	Z	160	160	10	5	10

Table 14-12
Search parameters for the K dump

The IDW2 estimates were similarly carried out in three passes, with a minimum of 5 and maximum of 10 samples used in the estimates and a limit of 2 samples per drillhole. The search volume applied to the Cc, Ec, Hc, Ic and Gc deposits was 60 mX by 60 mY by 6 mZ. The search volume was orientated at a 35° angle to mimic the angle of repose of the tailings, which tends to vary from 30° to 35°. The search was orientated by defining a centre line for each deposit, thereby dividing the deposit in half, where each half represents a dominant direction of deposition. The search ellipsoids were then orientated on either side to match this orientation. An example for the Hc block model is shown in Figure 14-9, where one side of the tailings is orientated at a 35° dip at an azimuth of 135°, while the other half is orientated at an azimuth of 315°.

Figure 14-9
Example of search ellipsoid orientation used for the Hc deposit



The search volume for the Hf and the Gf deposits was orientated horizontally as these deposits tend to be flat and extend over a larger footprint, lacking the high terraces observed in other deposits. The search ranges applied to the Hf and Gf deposits were 100 mX by 100 mY by 3 mZ and 80 mX by 80 mY by 3 mZ respectively.

14.8.1 Density

Density data coverage is limited to pits excavated to one metre below the surface of the tailings deposits. The unconsolidated nature of the material being sampled makes it impractical to take density measurements at depth. Density measurements were taken per material type, with these predominantly being either pegmatite, laterite or clay. Due to the limited data coverage, density could not be interpolated, therefore the average value per material type was assigned directly to the block model.

Density measurements were taken for all deposits except for the Ec tailings, where the average density of the pegmatite and tailings was calculated as the average for all density measurements from the eight deposits. The density assigned per material type for each deposit are summarised in Table 14-13.

Table 14-13
Average density assigned per material type for each deposit

Material Type	Cc	Ec	Gc	Gf	Hc	Hf	Ic	K
Pegmatite	1.61	1.56	1.55	1.55	1.54	1.54	1.66	1.54
Laterite	-	1.63	1.65	1.65	1.56	1.56	1.63	-
Clay	-	-	-	1.45	-	1.15	-	-

14.9 Validation of Estimates

The models were validated by:

- Comparison of the global estimate against the mean composite grades.
- Visual examination, in cross-section and plan, of the input data against the block model.
- Swath plot validation.

The mean grades of the block model for each domain were validated against the composite grades. Globally the estimated block grades compared favourably to the input data, with relative differences typically less than ten percent. Where larger percentage differences were observed, this typically translated to small relative differences in the mean values.

A comparison for each estimation domain, per deposit is presented in Table 14-14.

Table 14-14
Global mean comparison between capped composites and estimates

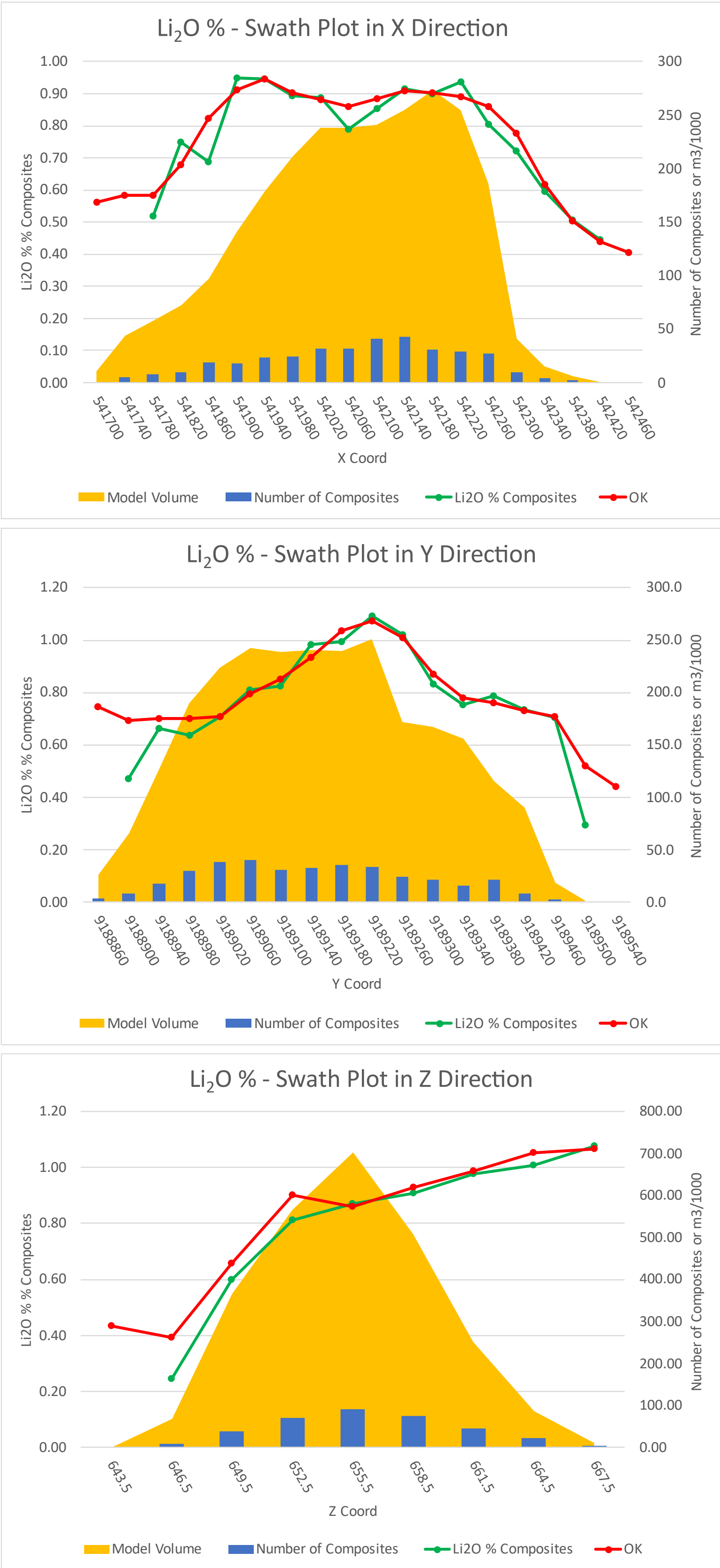
Assay	Domain	Composites			Block Model		Percentage Difference
		Number of Composites	Mean	CV	Mean	CV	
Cc Dump							
Li_2O %	PEG1	271	0.14	1.16	0.14	0.75	0%
Ec Dump							
Li_2O %	LAT1	38	0.06	0.32	0.06	0.13	-3%
	LAT2	30	0.05	0.27	0.05	0.13	0%
	PEG1	28	0.05	0.37	0.05	0.20	2%
	PEG2	87	0.07	0.35	0.07	0.23	1%
	PEG3	21	0.06	0.23	0.06	0.13	0%
Gc Dump							
Li_2O %	LAT1	19	0.04	0.46	0.04	0.18	-1%
	LAT2	338	0.04	0.65	0.04	0.36	-2%
	LAT3	2	0.04	0.51	0.04	0.00	0%
	PEG1	15	0.06	0.71	0.06	0.22	2%
	PEG2	85	0.24	1.39	0.24	1.42	0%
	PEG3	4	0.06	0.18	0.06	-	-
Sn ppm	LAT1	19	208	0.48	194.06	0.15	-6%
	LAT2	328	258	0.57	258.40	0.28	0%
	LAT3	2	88	0.11	87.50	0.00	0%
	PEG1	18	250	0.83	260.60	0.38	4%
	PEG2	83	382	0.69	330.18	0.46	-6%
	PEG3	4	210	0.25	210.00	-	-
Ta ppm	LAT1	19	21	0.52	19.84	0.13	-4%
	LAT2	328	21	0.63	20.05	0.30	-3%
	LAT3	2	9	0.01	8.85	-	-
	PEG1	18	13	0.52	12.18	0.24	-3%
	PEG2	83	22	0.51	19.80	0.34	-10%
	PEG3	4	21	0.26	21.35	-	-
	Gf Dump						
Li_2O %	CLA1	59	0.15	0.26	0.16	0.14	3%
	CLA2	16	0.12	0.27	0.12	0.20	0%
	LAT1	70	0.10	0.43	0.10	0.35	0%
	PEG1	81	0.24	0.81	0.25	0.57	2%
	PEG2	7	0.03	0.82	0.04	0.60	23%
Sn ppm	CLA1	59	142	0.25	144	0.15	2%
	CLA2	16	198	0.27	198	0.12	0%
	LAT1	70	146	0.36	151	0.18	3%
	PEG1	81	187	0.54	180	0.37	-4%
	PEG2	7	197	0.37	204	0.16	4%
Ta ppm	CLA1	59	17	0.26	16	0.17	-4%

Assay	Domain	Composites			Block Model		Percentage Difference
		Number of Composites	Mean	CV	Mean	CV	
	CLA2	16	23	0.17	23	0.10	1%
	LAT1	70	20	0.33	21	0.22	1%
	PEG1	81	24	0.51	24	0.36	2%
	PEG1	7	13	0.31	14	0.13	5%
	Hc Dump						
Li_2O %	LAT1	49	0.03	0.35	0.03	0.25	-4%
	LAT2	27	0.03	0.55	0.04	0.37	10%
	PEG1	10	0.04	0.32	0.04	0.18	2%
	PEG2	93	0.08	0.55	0.09	0.35	14%
	PEG3	13	0.04	0.11	0.04	0.04	2%
	Hf Dump						
Li_2O %	LAT1	28	0.03	0.47	0.03	0.28	2%
	PEG1	91	0.09	0.37	0.09	0.32	-2%
	Ic Dump						
Li_2O %	LAT1	140	0.04	0.46	0.04	0.28	-2%
	LAT2	129	0.04	0.70	0.04	0.48	2%
	PEG1	96	0.09	0.63	0.09	0.48	-2%
	PEG2	19	0.39	0.69	0.39	0.41	2%
Sn ppm	LAT1	140	370	0.61	369	0.32	0%
	LAT2	129	361	0.55	356	0.27	-1%
	PEG1	96	573	0.50	556	0.31	-3%
	PEG2	19	491	0.40	506	0.28	3%
Ta ppm	LAT1	140	13	0.44	13	0.25	-2%
	LAT2	129	14	0.40	14	0.22	3%
	PEG1	96	19	0.37	19	0.21	1%
	PEG2	19	25	0.41	26	0.27	4%
	K Dump						
Li_2O %	PEG1	237	0.66	0.38	0.67	0.25	2%
	PEG2	356	0.85	0.39	0.87	0.29	2%
Sn ppm	PEG1	226	653	0.40	656	0.21	1%
	PEG2	352	310	0.60	305	0.34	-1%
Ta ppm	PEG1	226	33	0.41	35	0.25	5%
	PEG2	352	25	0.41	25	0.21	0%

Due to the paucity of the data, the majority of the deposits did not lend well to being validated using swath plots, with the exception of the K, Hf and Gf deposits. For these deposits, swath plot validations in the X, Y and Z direction were used to locally validate the block estimates against the sample composites. No material biases in the estimates of the individual elements were identified. Examples of a swath plot validation for Li_2O for the K dump are shown in Figure 14-10.

Figure 14-10

Swath plot validation for Li_2O % for the K deposit



The block model was examined visually to ensure that the drillhole grades were locally well represented by the block model and it was found that the model validated reasonably well, with acceptable degrees of smoothing observed for all attributes. Examples of visual validation of the models for the K deposit in plan view and section are shown in Figure 14-11 and Figure 14-12 respectively.

Figure 14-11

K deposit estimated block model plan view– Li_2O %

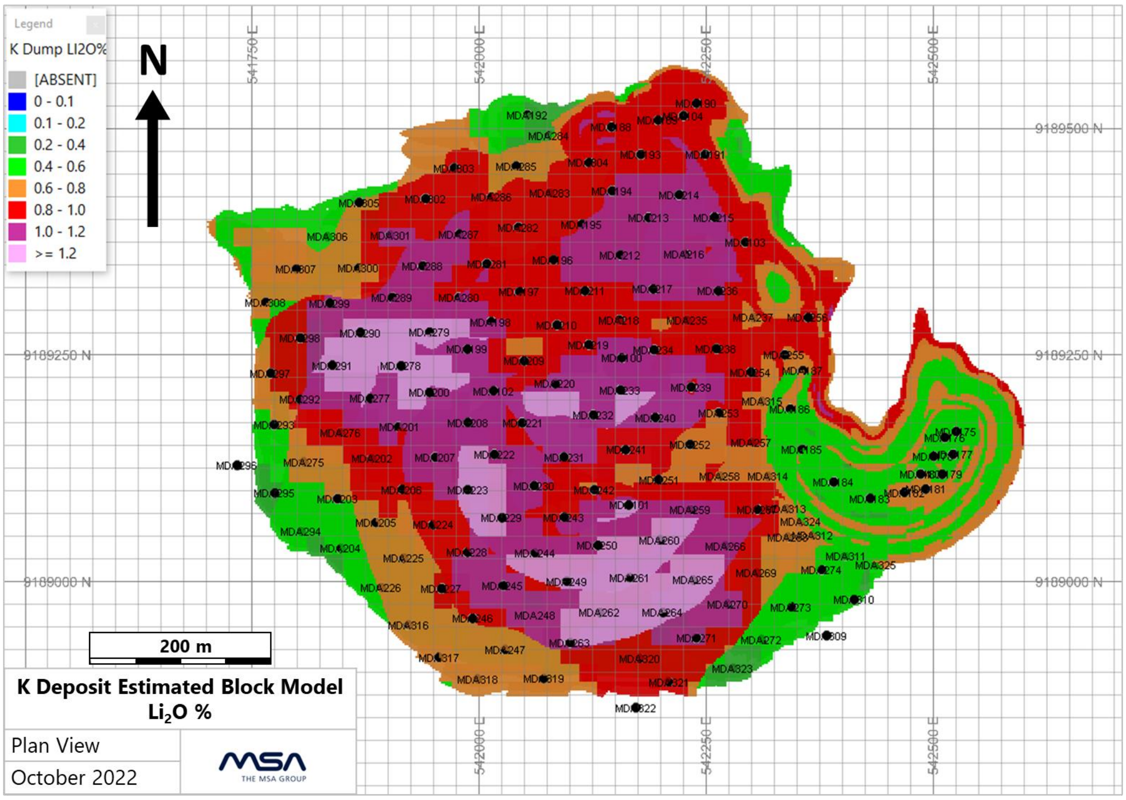
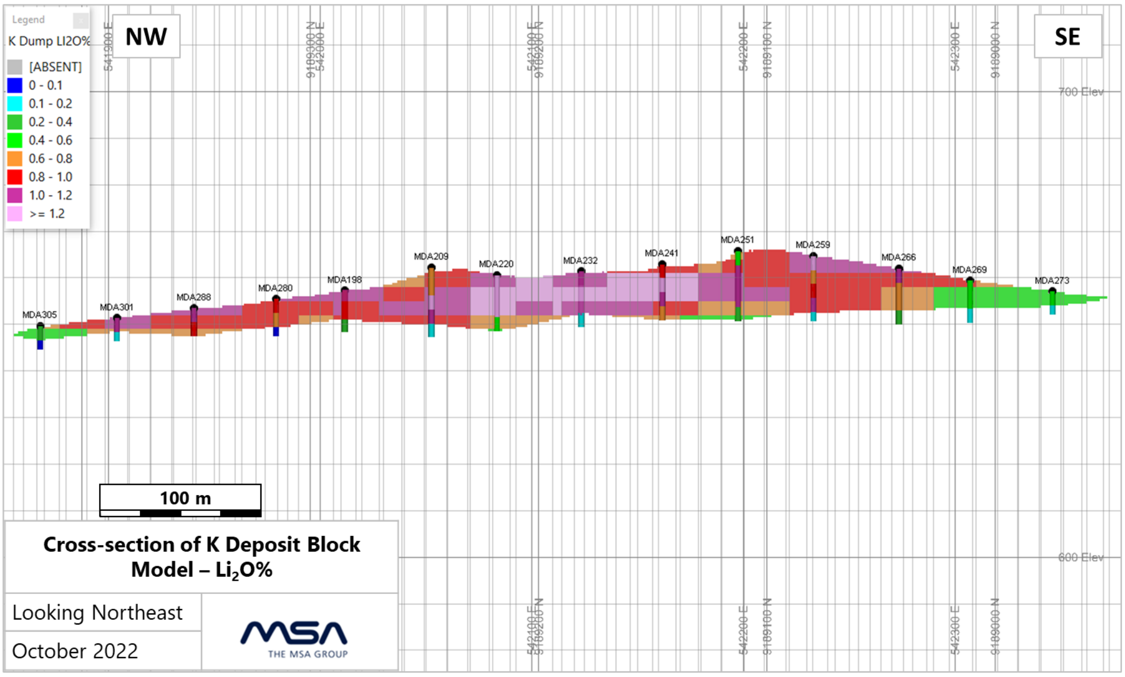
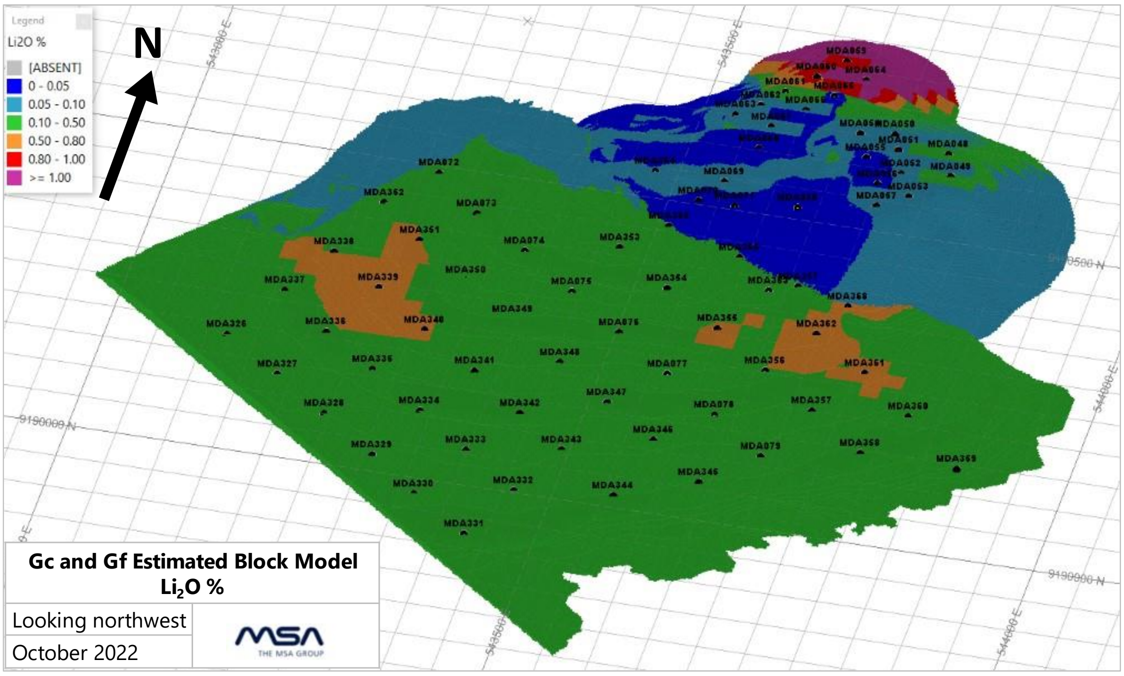


Figure 14-12
Cross-section through K deposit coloured on Li_2O % (looking northeast)



The Gc and Gf block models are illustrated in Figure 14-13.

Figure 14-13
Isometric view of the Gc deposit in the background and the Gf deposit (foreground)



14.10 Mineral Resource Classification

Classification of the Manono block models was based on the degree of geological uncertainty of the material types which constitute each tailings deposit, lithium grade continuity and variability and the frequency of the drilling data. The main considerations in the classification are as follows:

- The data that informs the Mineral Resource estimate has been collected using acceptable principles and the assays have been demonstrated to be of reasonable accuracy.
- The mineralisation shows reasonable lateral continuity within each tailings deposit.
- For the K deposit, the semivariogram ranges for lithium are 170 m, which is well within the drillhole spacing of 40 m for the lower lying material.

Given the aforementioned factors, the Manono Tailings Mineral Resources have been classified using the following criteria:

- The Mineral Resource was classified as Measured where the tailings deposit was homogenous in material type, drilled to a nominal 40 m grid spacing and where good continuity of Li_2O grades can be observed.
- Areas informed by drilling with a nominal grid spacing of 40 m to 80 m, with a maximum extrapolation of 40 m from the nearest drillhole were classified as Indicated Mineral Resources.
- Inferred Mineral Resources were classified where confidence in the estimates is low due to sparse drillhole coverage and where local estimates cannot be reliably made.

The Measured Mineral Resources for the Manono tailings are exclusively contained in the low-lying tailings material of the K deposit. The stacked tailings of the K deposit were classified as Inferred due to the sparse drillhole coverage. Achieving a dense drilling grid on the stacked tailings proved technically challenging due to the inability to safely drill this unconsolidated material and Tantalex is actively pursuing a way of drilling these tailings in order to increase the confidence in the estimates. The Indicated Mineral Resources are contained predominantly in the Hf deposit with a small portion present in the Gc deposit. The remainder of the Manono deposits were classified as Inferred due to sparse drillhole coverage.

The model classification is illustrated in Figure 14-14 for the K dump, Figure 14-15 for the Gf dump and Figure 14-16 for the Gc dump.

Figure 14-14
K deposit classification

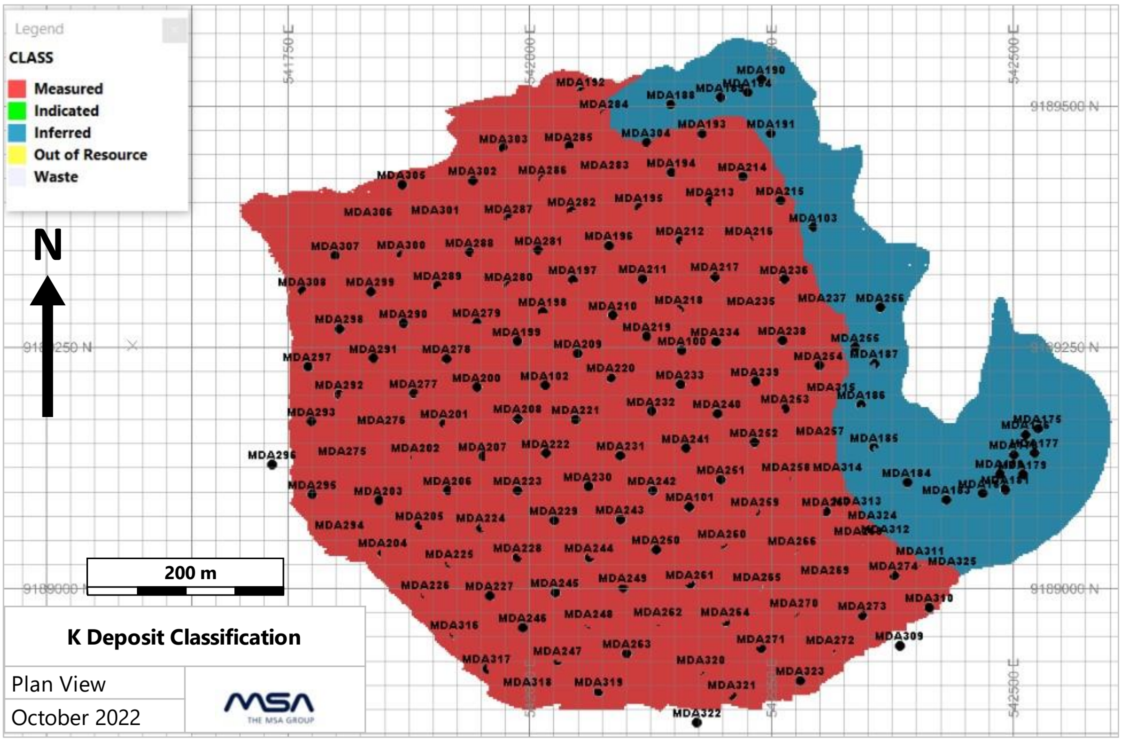


Figure 14-15
Gc deposit classification

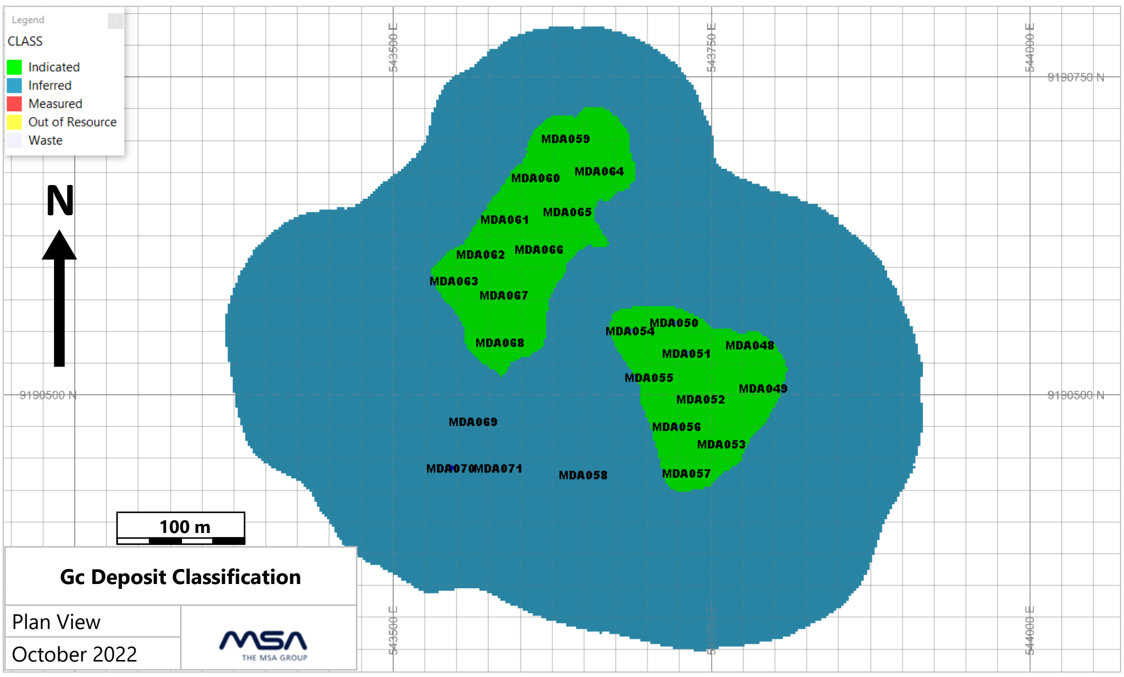
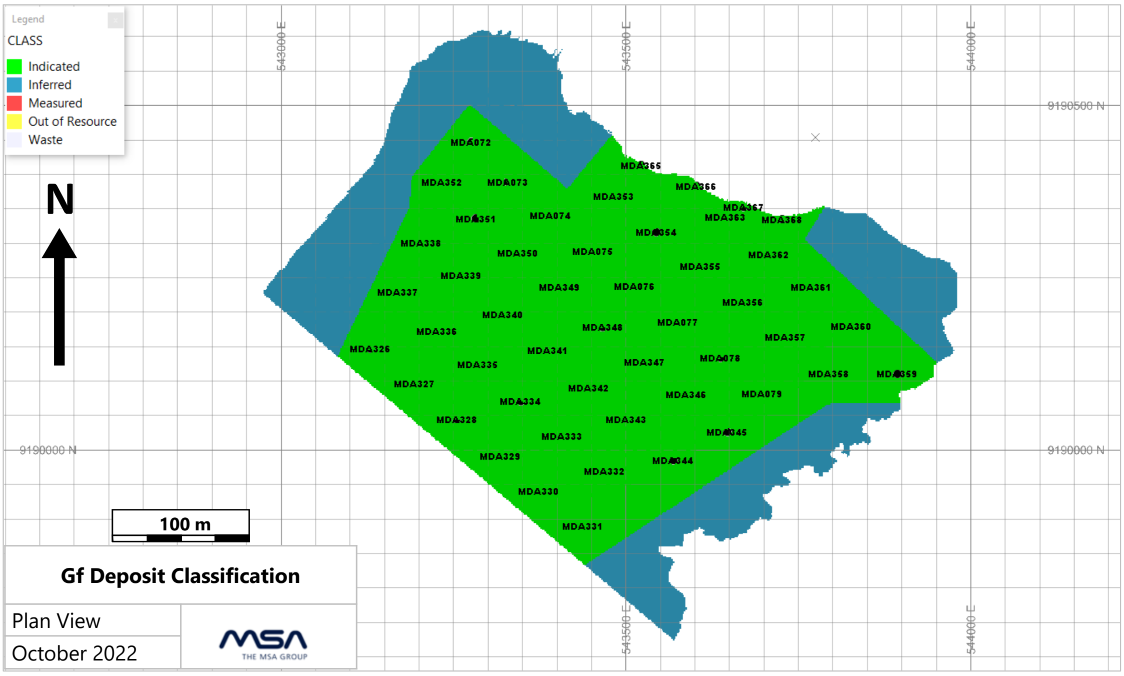


Figure 14-16
Gf deposit classification



14.11 Mineral Resource Statement

The Mineral Resource estimates as of 23 August 2023 are presented at a cut-off grade of 0.20% Li_2O for each deposit and totalled for each category in Table 14-15 for the Southern Sector deposits (Ic, Gc, Gf and K dumps). Due to the spatial arrangement of the high grade areas, which can be visually discerned from low-grade laterite areas, these deposits offer a sufficient degree of selectivity to be mined at the selected cut-off grade.

At the selected cut-off grade no Mineral Resources are reported for deposits Ec, Hc and Hf.

In the QP's opinion, the Mineral Resources reported herein at the selected cut-off grade have "reasonable prospects for eventual economic extraction", taking into consideration mining and processing assumptions.

Table 14-15
Manono Mineral Resources a 0.20% Li_2O cut-off grade – 23 August 2023

Deposit	Classification	Tonnes (Mt)	Li_2O %	Sn ppm	Ta ppm
Cc	Inferred	2.99	0.32	-	-
Ic	Inferred	0.51	0.49	583	29
Gc	Indicated	0.29	0.78	579	30
	Inferred	0.51	0.84	554	29
Gf	Indicated	1.39	0.35	183	22
	Inferred	0.13	0.33	209	26
K	Measured	3.77	0.86	306	25
	Inferred	2.33	0.67	656	35
Li_2O, Sn and Ta Mineral Resources					
Total	Measured	3.77	0.86	306	25
	Indicated	1.69	0.42	252	24
	Measured & Indicated	**5.46**	**0.73**	**289**	**25**
	Inferred	3.48	0.66	614	33
Li_2O only Mineral Resources					
Total	Inferred	2.99	0.32	-	-

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources are not Mineral Reserves, have no demonstrated economic viability*
3. *Li_2O % grades calculated by applying a factor of 2.153 to Li % grades*
4. *Mt = Million tonnes, ppm = parts per million*
5. *Inferred Li_2O, Sn and Ta Mineral Resources are totalled for the Southern Sector dumps (Ic, Gc, Gf and K).*
6. *Inferred Li_2O only Mineral Resources are for the Cc dump.*

14.11.1 Assessment of Reasonable Prospects for Eventual Economic Extraction (RPEE)

An assessment of Reasonable Prospects for Eventual Economic Extraction was undertaken based on costs provided by Tantalex and derived from the PEA. The following assumptions have been used to determine the cut-off grade and RPEE:

- Mining will be undertaken using bulldozers and loaders.
- Mining cost: USD 2.17 per tonne of rock
- Mining Recovery: 99%
- Process Recovery: 63% for Li_2O
- Revenue Royalty: 3%
- Payability: 98.5%

- Process Costs: 11.18 USD/tonne RoM
- Transport Costs: 361 USD/tonne of concentrate
- Indirect Costs including G&A: 76.5 USD/tonne of concentrate.
- Marketing Costs: 178.4 USD/tonne of concentrate
- Lithium Price: 2800 USD/tonne (SC6 – Spodumene Concentrate)

14.12 Comparison with Previous Estimate

The Mineral Resource estimate detailed in this reports represents the second Mineral Resource estimate reported for the Manono Lithium Tailings Project. The updated Mineral Resource estimate includes estimates for tin and tantalum which were previously excluded, as well as additional drilling for the Ic deposit.

A comparison for the total Mineral Resources between the previous estimate, with an effective date 13 December 2022, and the current estimate with an effective date 23 August 2023 is presented in Table 14-16.

Table 14-16
Manono Mineral Resource estimate compared with the 13 December 2022 Mineral Resource Estimate

Classification	13 December 2022				23 August 2023			
	Tonnes (Mt)	Li_2O %	Sn ppm	Ta ppm	Tonnes (Mt)	Li_2O %	Sn ppm	Ta ppm
Measured	3.77	0.86	-	-	3.77	0.86	306	25
Indicated	1.69	0.42	-	-	1.69	0.42	252	24
M&I	**5.46**	**0.72**	-	-	**5.46**	**0.73**	**289**	**25**
Inferred (Li_2O, Sn and Ta)	3.64	0.64	-	-	3.48	0.66	614	33
Inferred (Li_2O only)	2.99	0.32	-	-	2.99	0.32	-	-

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources are not Mineral Reserves, have no demonstrated economic viability*
3. *Li_2O % grades calculated by applying a factor of 2.153 to Li % grades*
4. *Mt = Million tonnes, ppm = parts per million*
5. *Inferred Li_2O, Sn and Ta Mineral Resources are totalled for the Southern Sector dumps (Ic, Gc, Gf and K).*
6. *Inferred Li_2O only Mineral Resources are for the Cc dump.*

15 MINERAL RESERVE ESTIMATES

Mineral Reserves have not been declared for the Manono Lithium Tailings Project.

16 MINING METHODS

Not applicable.

17 RECOVERY METHODS

Not applicable.

18 PROJECT INFRASTRUCTURE

Not applicable.

19 MARKET STUDIES AND CONTRACTS

Not applicable.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Not applicable.

21 CAPITAL AND OPERATING COSTS

Not Applicable.

22 ECONOMIC ANALYSIS

Not applicable.

23 ADJACENT PROPERTIES

The Manono project license for the in-situ pegmatite deposits (PR 13359) has been held 100% by Dathcom Mining SAS until very recently. As of the date of this report, the Mining Cadastre indicates that the licence is now back to being owned 100% by Cominiere SAS.

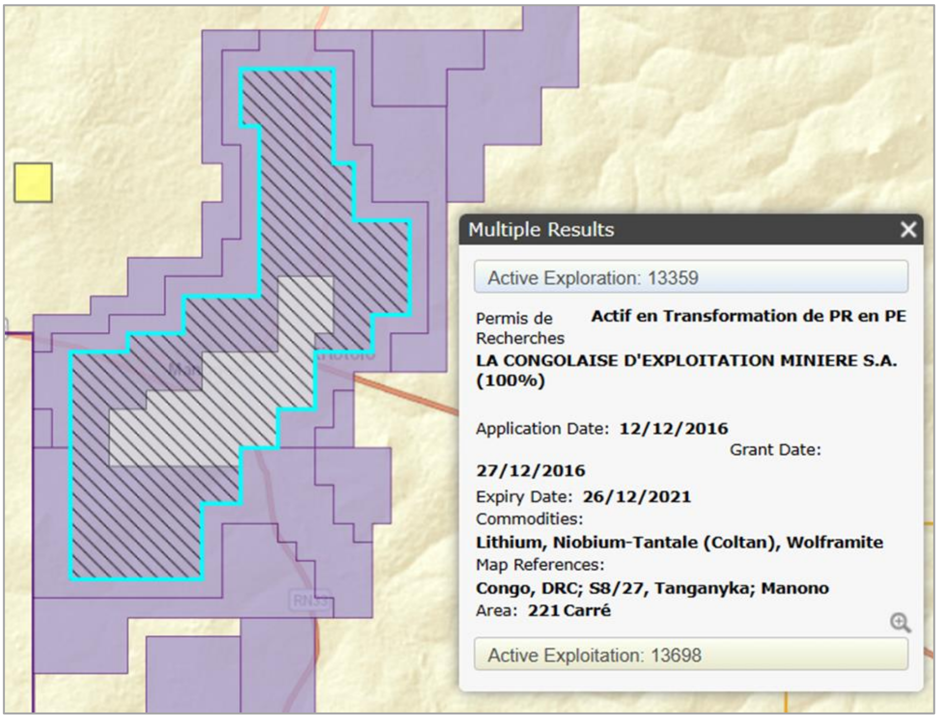
Dathcom was originally a Joint Venture held 30% by Cominiere SAS, 10% by Dathomir and 60% by AVZ Minerals. The current ownership of this Joint venture remains disputed and is currently the subject of litigation between the different Parties.

In 2020, Dathcom completed a Definitive Feasibility Study in which they reported a JORC (2012) Mineral Resource estimate as of 21 April 2020 of 269M tonnes in the Measured and Indicated and 131M tonnes in the Inferred category with an average grade of 1.65% Li_2O., 715 ppm Sn and 34 ppm Ta for the Manono Lithium and Tin Project (https://avzminerals.com.au/manono-mine).

The exploitation of this subsurface resource will eventually require the removal of some of the tailings from the tailings concession. Although these deposits do not have geological characteristics similar to those being reported, the expansion of these pits have an important bearing on the potential of Manono Lithium Tailings Project.

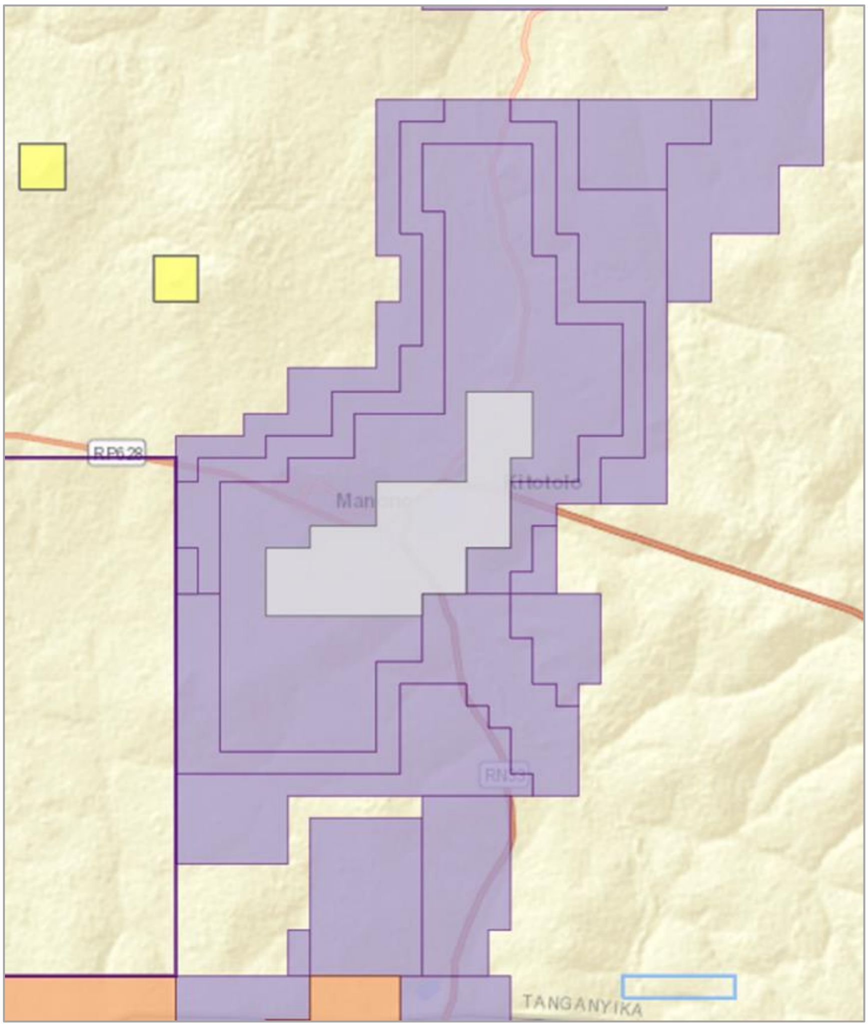
The Research Permits, PER4029 and PER4030, are owned by AVZ Minerals Congo SARLU (100%) granted on 21 July 2016 and expired on 20 July 2021 (Figure 23-1). The current status on the portal of the Mining Cadastre is as per Figure 23 2. (www.cami.cd)

Figure 23-1
Manono/Manono Extension Project license areas



Source: *AVZ Minerals (2022)*

Figure 23-2
Current Status on the portal of the Mining Cadastre



Source: *AVZ Minerals (2022)*

24 OTHER RELEVANT DATA AND INFORMATION

There is no additional information relevant to Geology and Mineral Resources

25 INTERPRETATION AND CONCLUSIONS

On behalf of Tantalex, MSA has completed a Mineral Resource estimate for the Manono tailings deposits. The Mineral Resources are based on aircore chips generated from a drilling programme which took place from September 2021 to July 2022.

The samples were subjected to a QAQC programme consisting of the insertion of CRMs, blank samples and the preparation of coarse duplicates. No significant contamination was identified and the CRM analysis suggests an acceptable degree of accuracy for all three elements. There is good internal and inter-laboratory precision for lithium, however the heterogeneous nature of the tin and tantalum mineralisation influences analytical precision which should be mitigated in the estimates by the use of a sufficient number of samples. The lithium grades were confirmed by a check assaying exercise but similar checks were not possible for tin and tantalum. The QP is satisfied that the assay results are of sufficient accuracy and precision for use in Mineral Resource estimation.

The estimates were constrained within modelled volumes representing the various material types making up each individual dump. Ordinary kriging was used to estimate the densely drilled K dump tailings, with the stacked material of the K and the other deposits being estimated using inverse distance squared. The models were validated by statistical and visual means and it was found that the estimates conformed to the data informing the estimates.

The Mineral Resources were reported in the Measured, Indicated and Inferred categories as shown in Table 14-15. The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines (2019) 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).

In the QP's opinion, the Mineral Resources reported herein at the selected cut-off grade have "reasonable prospects for eventual economic extraction", taking into consideration mining and processing assumptions. The Mineral Resource was reported at a cut-off grade of 0.20% Li_2O.

Table 25-1

Manono Mineral Resources a 0.20% Li_2O cut-off grade – 23 August 2023

Deposit	Classification	Tonnes (Mt)	Li_2O %	Sn ppm	Ta ppm
Cc	Inferred	2.99	0.32	-	-
Ic	Inferred	0.51	0.49	583	29
Gc	Indicated	0.29	0.78	579	30
	Inferred	0.51	0.84	554	29
Gf	Indicated	1.39	0.35	183	22
	Inferred	0.13	0.33	209	26
K	Measured	3.77	0.86	305	25
	Inferred	2.33	0.67	652	35
Li_2O, Sn and Ta Mineral Resources					
Total	Measured	3.77	0.86	306	25
	Indicated	1.69	0.42	252	24
	Measured & Indicated	**5.46**	**0.73**	**289**	**25**
	Inferred	3.48	0.66	614	33
Li_2O only Mineral Resources					
Total	Inferred	2.99	0.32	-	-

Notes:

1. *All tabulated data have been rounded and as a result minor computational errors may occur.*
2. *Mineral Resources are not Mineral Reserves, have no demonstrated economic viability*
3. *Li_2O % grades calculated by applying a factor of 2.153 to Li % grades*
4. *Mt = Million tonnes, ppm = parts per million*
5. *Inferred Li_2O, Sn and Ta Mineral Resources are totalled for the Southern Sector dumps (Ic, Gc, Gf and K).*
6. *Inferred Li_2O only Mineral Resources are for the Cc dump.*

At the selected cut-off grade of 0.2% Li_2O, no Mineral Resources are reported for the Ec, Hc and Hf deposits due to their low grade.

26 RECOMMENDATIONS

A strategy to drill the stacked tailings of the K deposit is currently being investigated, with the aim of providing sufficient data for higher confidence estimates for this material. The budgeted cost to complete this work is approximately 265,000 USD and includes drilling and assaying cost and consulting services to update the Mineral Resource estimates as detailed in Table 26-1.

It is in the QP's opinion that the proposed budget by Tantalex represents a reasonable cost estimate necessary to complete the above recommendations.

Table 26-1
Estimated cost of proposed program

Item	Total (USD)
Aircore drilling (3300 m)	$ 132 000
Assay (including shipping)	$ 44 000
Bulldozer	$ 8 000
Consulting Services	$ 46 435
Total (including 15% contingency)	**$ 265 000**

27 REFERENCES

AVZ Minerals Limited. ASX Announcement 21 April 2020. Accessed online on the 22/10/2022.

Dewaele, S., Goethals, H., and Thys, T. (2013). Mineralogical characterisation of cassiterite concentrates from quartz vein and pegmatite mineralisation of the Karagwe-Ankole and Kibara Belts, Central Africa. Geologica Belgica: 60, pp. 66-75.

Dewaele, S., Hulbosch, N., Cryns, Y., Boyce, A., Burgess, R., and Muchez, Ph. (2016). Geological setting and timing of the world class Sn, Nb-Ta and Li mineralization of Manono-Kitotolo (Katanga, Democratic Republic of Congo). Ore Geology Reviews: 72, pp. 373-390.

Ikigai Environmental Specialists. (2021). Final Volumes and Surveying Report of the Manono Tailings Dumps on behalf of Tantalex Resources Company.

Kinyaga, D. (2022). Tantalex Manono Dumps Bulk Density Measurement Report.

Kokonyangi, J. (2004). Structural constraints on cassiterite and columbite-tantalite mineralization in the Kibaran belt, D. R. Congo (Central Africa): implication for the timing of ore formation. Journal of Geosciences, Osaka City University, 2004: 47, Art. 11, pp. 127-140.

Kokonyangi, J.W., Kampunza, A.B., Armstrong, R., Yoshida, M., Okudaira, T., Arima, M. and Ngulube, D.A. (2006). The Mesoproterozoic Kibaride belt (Katanga, SE D.R. Congo). Journal of African Earth Sciences: 46, pp. 1-35.

Melcher, F., Graupner, T., Gäbler, H., Sitnikova, M., Henjes-Kunst, F., Oberthür, T., Gerdes, A. and Dewaele, S. (2013). Tantalum–(niobium–tin) mineralisation in African pegmatites and rare metal granites: Constraints from Ta–Nb oxide mineralogy, geochemistry and U–Pb geochronology. Ore Geology Reviews, 2013. (http://dx.doi.org/10.1016/j.oregeorev.2013.09.003)

Pohl, W.L., Biryabarema, M., Lehmann, B. (2013) Early Neoproterozoic rare metal (Sn, Ta, W) and gold metallogeny of the Central Africa Region: A review. Appl. Earth Sci: 122, pp. 66–82.

Scholtz, N. (2019). Manono Tailings Lithium and Tin Project, Democratic Republic of the Congo. NI43-101 Technical Report.

USGS (2010). Mineral-Deposit Model for Lithium-Caesium-Tantalum Pegmatites. Scientific Investigations Report 2010-5070-0.

https://avzminerals.com.au/manono-mine. Accessed on the 22/10/2022.

https://forrestgroup.com/en/manono-solar-power-plant-operational. Accessed on the 25/10/2022.

https://optron.com/trimble/products/r4s/. Accessed on the 25/10/2022.