Minesure Pty Ltd

Underground Mining Study

Mina do Barroso Project

Savannah Resources Ltd

[Status]

Nigel Spicer

Date:9 October 2019

Minesure Pty Ltd

Date: 9 October 2019

Company Savannah Resources Plc

Attention: Alan Rubio

Dear Alan

RE: Underground Mining Study

Please find enclosed our final report on the Underground Mining Study for the Mina do Barroso Project.

For and on behalf of Minesure Pty Ltd

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Table of Contents

EXECUTIVE SUMMARY ..... **i**

1 Introduction **1**

1.1 Scope of Work 1

1.2 Location 1

1.3 Project Description 1

2 Mining Options **2**

2.1 Orebody Characteristics 6

2.1.1 General Shape 6

2.1.2 Thickness 6

2.1.3 Plunge/Dip 7

2.1.4 Grade Distribution 7

2.2 Hanging Wall and Foot Wall Geotechnics 7

2.2.1 Rock Strength 7

2.2.2 Fracture Spacings 8

2.2.3 Fracture Strength 8

2.3 Ore Zone Geotechnics 9

2.3.1 Rock Strength 9

2.3.2 Fracture Spacings 9

2.3.3 Fracture Strength 9

2.4 Results matrix 9

3 Mining Description **11**

3.1 Long Hole Stoping 11

3.2 Decline Access 12

3.3 Orebody Access 12

3.4 Drilling and Blasting 13

3.5 Ore Load Out 13

3.6 Haulage 14

3.7 Backfilling 14

3.8 Productivity estimation 14

4 Potential Mining Inventory Estimation **15**

4.1 Cut-Off Grade Estimation 15

4.2 Estimated Inventory 16

5 Project Cost Estimation **17**

5.1 Capital Costs 17

5.1.1 Mobilisation and Establishment 17

5.1.2 Infrastructure 17

5.1.3 Portal Excavation 17

5.1.4 Development – Waste 17

5.1.5 Ore Definition 18

5.1.6 Backfilling 18

5.1.7 Demobilisation 18

5.1.8 Summary of Capital Costs 18

5.2 Operating Costs 19
5.2.1 Development 19
5.2.2 Production 19
5.2.3 Backfilling 19
5.2.4 Owners Team 19
5.2.5 Summary of Operating Costs 20
5.3 Estimated Capital and Operating Costs 20
6 Cashflow Analysis 20
6.1 Revenue 20
6.2 Non-Mining Costs 21
6.3 Project Cashflow Estimation 21

List of Tables

Table 1 Estimated Capital Costs iv
Table 2 Estimated Operating Costs iv
Table 3 Estimated Capital and Operating Costs v
Table 4 Revenue Estimation v
Table 5 Non-Mining Cost Factor Estimation vi
Table 6 Project Cashflow Estimation vi
Table 2.1 –Orebody Characteristics 4
Table 2.2 –Hanging Wall Geotechnics 4
Table 2.3 –Ore Zone Geotechnics 5
Table 2.4 –Footwall Geotechnics 5
Table 2.5 –Rock Strength Classification 8
Table 2.6 –ATV Discontinuity Survey Results 8
Table 2.7 –Method Selection Matrix Results 10
Table 4.1 Cut-Off Grade Estimation 15
Table 5.1 Estimated Capital Costs 18
Table 5.2 Owners Team 19
Table 5.3 Estimated Operating Costs 20
Table 5.4 Estimated Capital and Operating Costs 20
Table 6.1 Revenue Estimation 20
Table 6.2 Non-Mining Cost Factor Estimation 21
Table 6.3 Project Cashflow Estimation 22

List of Figures

Figure 1 – Mina do Barroso Project Location 1
Figure 2 – Grandao Lodes and Final Open Pit Design 6
Figure 3 – Cross Section through Grandao Lodes 7
Figure 4 – Schematic of Long Hole Stoping with Backfill 11
Figure 5 – Section through Orebody Development 13
Figure 6 – Average Unit Mining Costs 16
Figure 7 – Estimated Annual Mill Feed Grades Li2O% 21
Figure 8 – Estimated Annual and Cumulative Cashflow ($) 22

EXECUTIVE SUMMARY

Introduction

Minesure Pty has been requested by Savannah Resources Plc to compile a study evaluating the viability of mining the Grandao orebody by underground methods.

The Mina do Barroso project is located in the parish of Covas do Barroso, municipality of Boticas, district of Vila Real, between the Serra do Barroso and the river Tâmega, about 12 km to Southwest Boticas in the northern part of Portugal**.**

Exploration drilling has identified a resource of 17.7Mt at 1.1 % Li2O for a cut-off grade of 0.5% Li2O. In 2018 Hatch compiled a scoping study to determine the viability of exploiting the deposit by open pit means. The Deposit is near surface and amenable to open pit mining.

Mining Methods

Nicholas (1981) devised a system to numerically assess which underground mining method would give the optimum results. The system ranked mining methods according to the following criteria:

- Orebody Characteristics
- Hanging Wall geotechnics
- Ore zone geotechnics
- Footwall geotechnics

A system using cut and fill stoping methods was deemed the most appropriate based on Nicholas' criteria. This method involves mining out sections of the orebody as "stopes" and then backfilling the void with cemented waste material. Typically, the dimension of each stope could be 20m wide, 30m – 40m high and ~50m long, depending on the orebody geometry. The ore within a stope is extracted using long hole drilling and blasting and when all the ore has been recovered the void is backfilled with waste material

The main lode at Grandao has a true thickness of 20m however the vertical height is approximately 30m which is very amenable to long hole open stoping (LHOS).

Decline Access

The topography above the lodes is steeply dipping from east to west. The rock above the proposed first development level is somewhat oxidised and probably too unstable to accommodate underground mining. In order to gain initial access to the mineralised zones a slightly inclined drift will be mined from the surface west of the ore zone to a position on the footwall of the lode. From that point a decline would be mined in the footwall. Because the lode dips at about 30° the access ramp would consist of elongated loops rather than a spiral so as to follow the same dip. The decline would have a gradient of -14% (-1 in 7) and would have a profile of 6.5m by 5.5m which would accommodate thirty tonne all-wheel drive (AWD) trucks.

A ventilation rise would be mined close to the decline to provide ventilation. Crosscuts would be mined off the decline to intersect the rise

Orebody Access

Horizontal crosscuts would be mined from the decline at 30m vertical intervals to intersect the orebody footwall. A horizontal waste foot wall drive would be mined to parallel the orebody footwall approximately ten metres away from the contact over its entire strike length. The purpose of this drive is initially to provided permanent access to extract the ore for the level above and subsequently to emplace the waste backfill.

Crosscuts would be mined from the footwall drive through the orebody at 50m horizontal intervals to determine the footwall and hanging wall positions. An ore drive would then be mined in the centre of the orebody along its entire length. These ore drives will provide access to mine out the stope above. Where the width of the orebody exceeds thirty metres is necessary to split the stope and mine it as two separate entities.

The system of drives and crosscuts is repeated at the mid height of the stope and the base of the stope. The ore drive which is at the mid height of the stope becomes the drilling drive for the long holes and the drives at the bottom are used to extract the broken ore.

Drilling and Blasting

Drilling of the long holes would be carried out by specialist drill rigs which are computer controlled to ensure accuracy of placement of holes. A drill pattern consists of a series of rings of holes drilled radially from within the drilling drive and extend out to the perimeters of the stope. The number of holes required will depend on the hardness of the rock.

Drill rings are blasted sequentially to slice off the ore along strike however a free face needs to be established before the first ring blast takes place. The free face (slot) is established by drilling a slot rise using the long hole drill. The slot rise consists of concentric circles of holes drill up and down dip around a central “cut” hole. The concentric holes around the cut hole are charged and blasted sequentially to create a rise. The rise is further enlarged by drilling and blasting holes which widen the slot out to the full width of the orebody.

Ore Load Out

The blasted ore falls to the bottom loading level of the stope. A front-end loader is used to recover the ore and it is loaded directly into a truck or stockpiled into a “cuddy” for later removal. The loaders will be capable of being operated remotely as it will be necessary for the loader to enter the open stope.

Haulage

Ore from the stopes will be loaded into thirty tonne capacity trucks and hauled to the ROM pad. Waste derived from the development activities will be stockpiled underground where possible or stockpiled on the surface for later reclamation. It is proposed that a large portion, if not all, of the waste will be backfilled underground.

Backfilling

To maintain ground stability and maximise ore recovery it will be necessary to fill the stoped-out voids with a cemented fill. This supports the hanging wall and provides a solid roof for work to continue underneath depleted stopes. It is envisaged that the fill will comprise a mixture of cement, tailings and waste rock. This will need to be mixed in a batching plant and delivered underground as a backload by the haulage trucks. Some infrastructure will need to be built to deslime the tailings, crush the rock to a suitable size and mix the cement to the required specifications. It is anticipated that backfilling will take place concurrently with ore production using different stopes.

Productivity estimation

There are between ten and twelve stopes planned for each level and it is proposed to mine on at least two levels at once. Given the number of faces available it should be possible to extract from three stopes on one level at once. This would give a potential annual production rate of 570,000 tonnes. There are also smaller lodes which will be mined at the same time which could increase the annual production rate up to 750,000tpa and this target will be used for scheduling purposes.

Potential Mining Inventory Estimation

The quantum and geometry of the inventory is determined by the distribution of grade through the orebody and the amenability of solids to be economically extracted.

Cut-off Grade Estimation

The economic cut-off grade is determined from the various factors such as cost of production and product price. Most of the costs and factors are derived from the April 2019 update of the scoping study. The underground mining cost used in estimating the cut-off grade is based on benchmarking of unit mining costs of several similar underground operations. It is estimated that the cut of grade would be 1% Li2O.

Estimated Inventory

Inferred resources have been included in this estimation on the basis that they will show the potential of the project to extend to depth. Whilst the orebody outcrops at surface the top of the orebody has oxidised to a variable extent down to the 490m Rl. It is unlikely that this portion can be mined safely by underground techniques so it is proposed to leave it intact and recover it by open pit methods. Using the vertical limits explained above, the 1.0% li2O cut-off grade and only JORC resources the undiluted inventory is estimated to be 7.6Mt at 1.24% Li2O.

Project Cost Estimation

Capital Costs

The major cost items in underground mining, assuming contractor mining, are shown in Table1:

Table 1 Estimated Capital Costs	
Item	**$M**
Mobilisation and Establishment	2.0
Development	43.9
Plant and Equipment	2.0
Ore Definition	1.0
Backfill Plant	2.5
Demobilisation	0.7
Total	**52.0**

Operating Costs

Operating costs have been estimated based on recent contractor prices for a similar sized project. Costs associated with waste mining have been capitalised. Costs associated with development in ore and stoping are deemed to be operating.

Approximately 12,700m of horizontal ore development is required. This will produce 840kt of ore which is trucked to the surface. The majority of the 6.7Mt of ore is produced by stoping activities. This requires drilling of 1.5 million linear metres of long holes over the 11year mine life.

Backfill material will have 5% cement component so as to provide it with sufficient strength to provide a safe working area. It is estimated that this will cost $15M over the life of project.

The owner's team will provide supervision and technical support and comprise twelve people to provide 24/7 cover. The associated cost is $1.0M per year.

The quantities and cost of operating activities are shown in Table 2.

Table 2 Estimated Operating Costs			
Item	Unit	Quantity	Cost $M
Development	Km	12.7	235.0
Production	Mt	6.7	100.3
Backfill	Mt	6.0	15.0
Total			350.2

Capital and Operating Costs

The total estimated Capital and Operating costs are shown in Table 3

Table 3 Estimated Capital and Operating Costs		
Item	$M	Unit cost $/t ore
Capital	52.0	6.90
Operating	350.2	46.70
Total	402.2	53.60

Cashflow Analysis

An annual cashflow model has been prepared using the costs and productivities estimated in this study and those used in the April 2019 cashflow model.

Revenue

There is a gross state royalty of 3% payable for all shipped spodumene. The price of spodumene (6% Li2O) used in this study is $679/t for which $20.37 is subtracted for the royalty. The estimated metallurgical recovery is 80% which means that 20% of the lithium that is fed into the plant passes through to the tailings dam. In effect the company receives $527 for each tonne of spodumene that is fed to the processing plant. This calculation is shown in Table 4.

Table 4 Revenue Estimation		
Item	Unit	Value
Spodumene Price (6%)	$	679
State Royalty	%	3
Metallurgical Recovery	%	80
Realised Value	$/t Spodumene	526.90

Non-Mining Costs

Cost factors used in the cashflow estimation are shown in Table 5. These were derived from the April 2019 cashflow model.

Table 5 Non-Mining Cost Factor Estimation	
Item	Value $/t
Treatment Cost	18.29
Fixed Royalty	0.01
Downstream Product Handling	8.17
Administration Cost	4.24
Total	30.71

Project Cashflow Estimation

The revenue and cost factors were used to estimate the cashflow resulting from the underground mining of the Mina Do Barroso project. This is shown annually in Table 6.

Table 6
Project Cashflow Estimation

Item	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Treatment Cost (Li)	137.2	1.0	13.6	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	12.9	0.0
Fixed Royalty	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Downstream Ore Handling (Li)	61.3	0.5	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	5.8	0.0
Admin Cost	31.8	0.2	3.1	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.0	0.0
Mining Opex	350.2	17.9	36.4	34.8	35.8	33.0	32.8	33.5	33.4	32.9	30.9	28.9	0.0
Total Operating Costs	580.6	19.6	59.2	57.8	58.8	56.0	55.8	56.6	56.5	55.9	53.9	50.5	0.0
Capital Costs	52.0	16.2	7.4	4.7	4.7	4.4	3.3	5.1	2.8	2.8	0.0	0.0	0.7
Total Costs	632.7	35.8	66.6	62.5	63.5	60.4	59.1	61.7	59.2	58.7	53.9	50.5	0.7
Revenue	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
k't Spodumene recovered	1,127	8	113	114	113	113	114	114	112	111	110	103	0
Value $M	593.6	4.4	59.5	59.9	59.6	59.5	59.9	60.2	59.2	58.7	58.2	54.4	0.0
Cashflow $M	-35.1	-27.4	-7.2	-2.6	-3.9	-1.0	0.8	-1.5	-0.1	0.0	4.3	3.9	-0.7

The estimation indicates that mining of the project would result in a negative cashflow of $35.1M.

1 INTRODUCTION

1.1 Scope of Work

Minesure Pty has been requested by Savannah Resources Plc to compile a study evaluating the viability of mining the Grandao orebody by underground methods.

1.2 Location

The Mina do Barroso project is located in the parish of Covas do Barroso, municipality of Boticas, district of Vila Real, between the Serra do Barroso and the river Tâmega, about 12 km to Southwest Boticas in the northern part of Portugal**.**

Figure 1

Mina do Barroso Project Location

1.3 Project Description

Exploration drilling has identified a resource of 17.7Mt at 1.1 % Li2O for a cut-off grade of 0.5% Li2O. In 2018 Hatch compiled a scoping study to determine the viability of exploiting the deposit by open pit means. The Deposit is near surface and amenable to open pit mining.

2 MINING OPTIONS

There are several methods which can be utilised to extract ore by underground mining. Nicholas (1981) devised a system to numerically assess which method would give the optimum results. The system uses the following information to rank available methods:

- Orebody Characteristics
  - Shape
    - Massive
    - Tabular
    - Irregular
  - Thickness
    - Narrow
    - Intermediate
    - Thick
    - Very Thick
  - Plunge
    - 0°-30°
    - 30° - 60°
    - >60°
  - Grade Distribution
    - Uniform
    - Gradational
    - Erratic
- Hanging Wall geotechnics
  - Rock strength
    - Weak
    - Moderate

        - Strong
    - Fracture Spacing
        - Very close
        - Close
        - Wide
        - Very wide
    - Fracture Strength
        - Weak
        - Moderate
        - Strong
- Ore zone geotechnics
    - Uses the same sub-categories as the Hanging Wall geotechnics
- Footwall geotechnics
    - Uses the same sub-categories as the Hanging Wall geotechnics

Nicholas assigned each variable a value for each method which denoted its appropriateness to the method. A low number indicated that it was not very suitable and vice versa. The categories and assigned values are shown in Tables 2.1 to Table 2.4. Methods which were totally unsuitable were assigned a value of -49. The highest assigned value is four.

Table 2.1
Orebody Characteristics

Method	General Shape			Ore Thickness				Ore Plunge (Dip)			Grade Distribution		
	Massive	Tabular	Irregular	Narrow	Intermediate	Thick	Very Thick	0 - 30	30 - 60	+60	Uniform	Gradational	Erratic
				<5m	>5m, <30m	>30m, <50m	>50m						
Open Pit	3	2	3	2	3	4	4	3	3	4	3	3	3
Block Caving	4	2	0	-49	0	2	4	3	2	4	4	2	0
Sub-level Stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sub-level Caving	3	4	1	-49	0	4	4	1	1	4	4	2	0
Longwall Mining	-49	4	-49	4	0	-49	-49	4	0	-49	4	2	0
Room & Pillar	0	4	2	4	2	-49	-49	4	1	0	3	3	3
Shrinkage Stoping	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut & Fill Stoping	0	4	2	4	4	0	0	0	3	4	3	3	3
Top Slicing	3	3	0	-49	0	3	4	4	1	2	4	2	0
Square set Stoping	0	2	4	4	4	1	1	2	3	3	3	3	3

Table 2.2
Hanging Wall Geotechnics

Method	Rock Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Wide	Very Wide	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	2	1	3	4	3	0	4	2	0
Sub-level Stoping	-49	3	4	-49	0	1	4	0	2	4
Sub-level Caving	3	2	1	3	4	3	1	4	2	0
Longwall Mining	4	2	0	4	4	3	0	4	2	0
Room & Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	4	2	1	4	4	3	0	4	2	0
Cut & Fill Stoping	3	2	2	3	3	2	2	4	3	2
Top Slicing	4	2	1	3	3	3	0	4	2	0
Square set Stoping	3	2	2	3	3	2	2	4	3	2

Table 2.3 Ore Zone Geotechnics										
	Rock Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Wide	Very Wide	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	1	1	4	4	3	0	4	3	0
Sub-level Stoping	-49	3	4	0	0	1	4	0	2	4
Sub-level Caving	0	3	3	0	2	4	4	0	2	2
Longwall Mining	4	1	0	4	4	0	0	4	3	0
Room & Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	1	3	4	0	1	3	4	0	2	4
Cut & Fill Stoping	3	2	2	3	3	2	2	3	3	2
Top Slicing	2	3	3	1	1	2	4	1	2	4
Square set Stoping	4	1	1	4	4	2	1	4	3	2

Table 2.4 Footwall Geotechnics										
Method	Rock Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Wide	Very Wide	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	2	3	3	1	3	3	3	1	3	3
Sub-level Stoping	0	2	4	0	0	2	4	0	1	4
Sub-level Caving	0	2	4	0	1	3	4	0	2	4
Longwall Mining	2	3	3	1	2	4	3	1	3	3
Room & Pillar	0	2	4	0	1	3	3	0	3	3
Shrinkage Stoping	2	3	3	2	3	3	2	2	2	3
Cut & Fill Stoping	4	2	2	4	4	2	2	4	4	2
Top Slicing	2	3	3	1	3	3	3	1	2	3
Square set Stoping	4	2	2	4	4	2	2	4	4	2

2.1 Orebody Characteristics

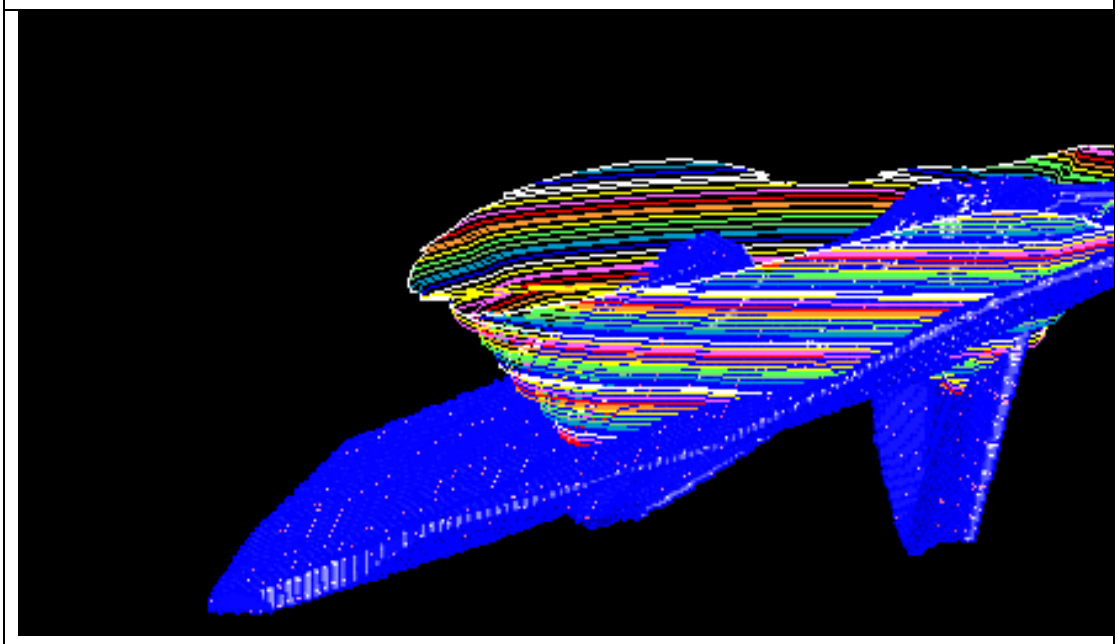
Observations relating to the orebody geometry were based on advice received from the Exploration Manager. It should be noted that there will be marked variations in discrete areas however overall the Grandao orebody is reasonably consistent.

2.1.1 General Shape

Of the three categories given in Table 2.1, the orebody can be considered generally tabular in form. As shown in Figure 2 there are two separate lodes comprising the Grandao orebody:

- Main lode which dips at approximately 30° to the west, is fairly uniform in thickness and continuity, and includes the majority of the resource. There is also a smaller lode which lies in the footwall of the main lode.
- East lode, which is essentially vertical, comparatively narrow but uniform in thickness.

Figure 2
Grandao Lodes and Final Pit Design



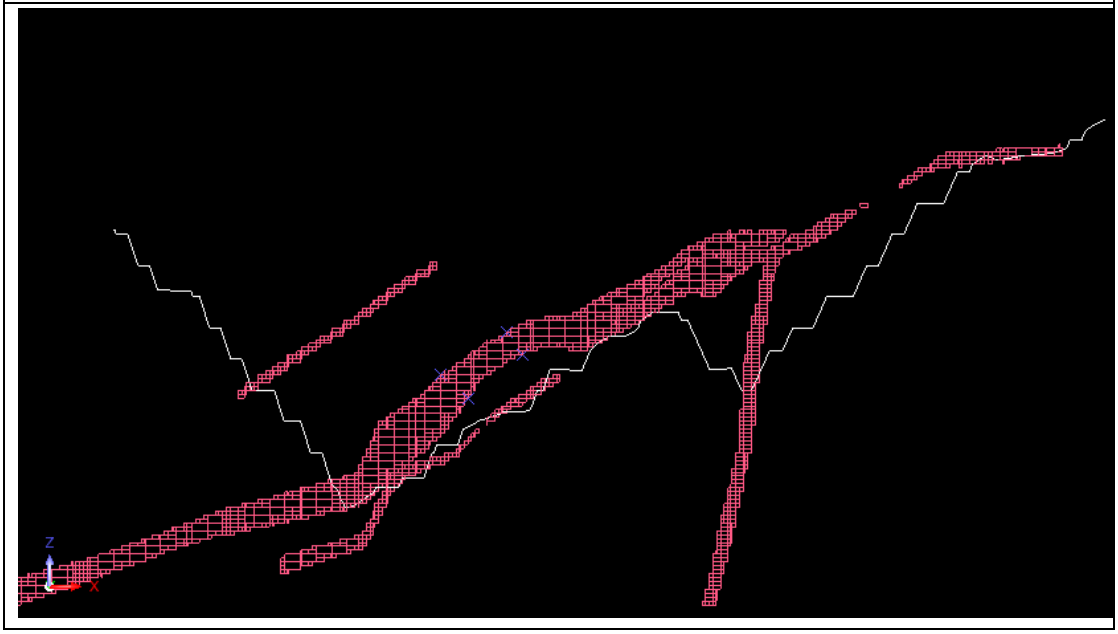
2.1.2 Thickness

A cross section through the Grandao resource model, Figure 3, shows that the main lode has a true thickness of between 16m and 20m. The east lode averages between eight metres and ten metres.

Based on the categories in Table 2_1 the main lode would be of intermediate thickness and the east lode would be considered narrow.

Figure 3

Cross Section through Grandao Lodes



2.1.3 Plunge/Dip

As shown in Figure 3 the lodes have very different dips. The main lode dips at approximately 30° whilst the east lode is vertical.

2.1.4 Grade Distribution

There are significant variations to the distribution of grade within the lodes, ranging from 0.8%Li2O up to 1.8% Li2O on a single flitch. There are also variations in the vertical plain. However, the economic mineralisation is generally contained within the defined lode systems which would classify the distribution as uniform.

2.2 Hanging Wall and Foot Wall Geotechnics

Knight Piesold Compiled a geotechnical report recommending the slope design parameters for open pit operation. The report did not differentiate between footwall and hanging wall domains. For this study it is assumed that the footwall and hanging wall rocks, both of which are schistose, are of similar nature.

2.2.1 Rock Strength

The rock strength is measured by undertaking unconfined compressive strength (UCS) testing in which a core of rock is subjected to compressive force until it fails. The force required to induce failure is measured and this becomes the UCS of the rock specimen and is measure in Mega Pascals (Mpa). The average UCS of the fresh schist samples, which were measured as part of the study, was 60MPa. Weathered specimens had lower UCS however these are discounted in this study as the majority of mining will be undertaken in fresh conditions.

Rock can be classified according to its UCS as shown in Table 2_5.

Table 2.5 Rock Strength Classification	
Classification	**UCS (MPa)**
Very Strong	>100
Strong	50-100
Moderately Strong	12.5-50
Moderately Weak	5 – 12.5
Weak	<5

The table indicates that the schist is in the “strong” rock classification.

2.2.2 Fracture Spacings

Fractures within a rock fabric are weaknesses which reduce the integrity of the rock proportionally to the frequency. The more fractures there are then the weaker the rock. This has implications in the hanging wall but is far less important in the footwall. With any underground mining technique, it is important that the hanging wall stays intact whilst ore is being extracted because failure leads to ore loss and high dilution. Table 2.6 is a modified form of Table 2.2 from the Knight -Piesold report showing the results of acoustic televiewer survey conducted on three boreholes.

Table 2.6 ATV Discontinuity Survey Results				
Hole ID	ATV From (m)	ATV to (m)	ATV Discontinuities	Frequency (m)
18GRARC059	32.4	54.6	10	2.2
18GRARC122	110.5	192.4	129	0.6
18GRARC030	16.0	66.4	27	1.9

In the first hole the average separation between fractures is 2.2m. Overall the results indicate that the rock is fractured but not excessively so. On this basis the fractures could be considered to be “close”

2.2.3 Fracture Strength

The Knight Piesold report does not refer to the strength of the fractures. This factor is a measure of the cohesion between fractures to indicate if they can slide easily or not. Fractures which have an infill of clay would readily part if exposed during mining however fractures which are devoid of foreign material would tend to remain intact. As there is little evidence of any deleterious infill material it will be assumed that the rock fracture strength will be moderate.

2.3 Ore Zone Geotechnics

2.3.1 Rock Strength

Testing of pegmatite cores indicated an average UCS of 136MPa which places the orebody in the strong category.

2.3.2 Fracture Spacings

There is little information specific to the pegmatite consequently the spacing will be considered to be the same as for the hanging wall and footwall.

2.3.3 Fracture Strength

There is little information specific to the pegmatite consequently the spacing will be considered to be the same as for the hanging wall and footwall. This may be conservative as the strength would be inherently greater due to the higher rock strength.

2.4 Results matrix

The relevant factors in Table 2.1 to 2.4 are inserted into a matrix and totalled for each mining method. The method with the highest value is deemed the most suitable for the stated conditions. The results are shown in Table 2.7.

The sum of the factors is shown in the column on the right had side. The two methods with the highest score are “Cut and Fill Stoping” and “Square Set Stoping”. Square set stoping involves maintaining open stopes using large baulks of timber which is not efficient, cost effective or environmentally desirable. Setting sets is a time consuming and labour-intensive operation which would slow down production. Timber sets are also liable to fail under extreme conditions.

Consequently, the chosen method of extraction will be Cut and Fill stoping.

Table 2.7

Method Selection Matrix Results

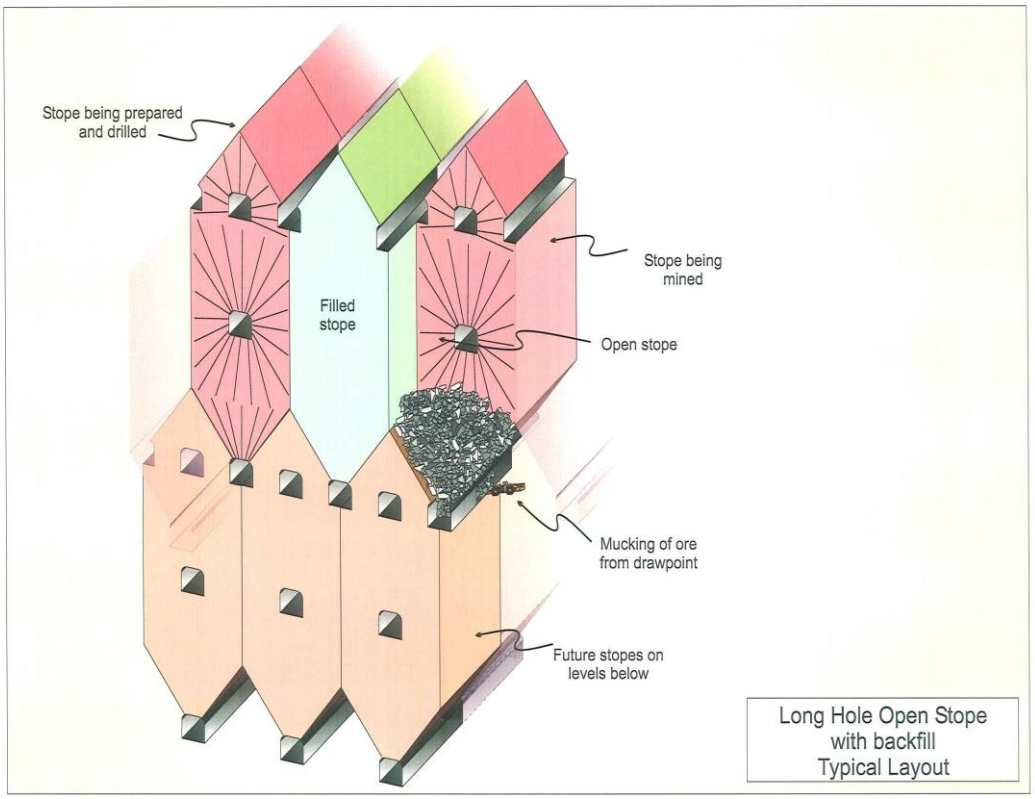
Method	Geometry				H/W Geotech			Ore Geotech			FW Geotech			
	Shape	Thick-ness	Plunge	Grade Dist	Rock strength	Fracture		Rock strength	Fracture		Rock strength	Fracture		Total
						Spacing	Strength		Spacing	Strength		Spacing	Strength	
Block Caving	2	0	3	2	1	5	3	1	5	4	4	4	4	39
Sub-level Stoping	2	2	2	3	5	0	3	5	0	3	5	0	1	32
Sub-level Caving	4	0	1	2	1	5	3	4	3	3	5	1	3	35
Longwall Mining	4	0	4	2	0	5	3	0	5	4	4	3	4	38
Room & Pillar	4	2	4	3	5	1	3	5	1	3	5	1	4	42
Shrinkage Stoping	2	2	2	2	1	5	3	5	1	3	4	4	3	37
Cut & Fill Stoping	4	4	0	3	3	4	4	3	4	4	3	5	5	46
Top Slicing	3	0	4	2	1	4	3	4	1	3	4	4	3	36
Square set Stoping	2	4	2	3	3	4	4	1	5	4	3	5	5	46

3 MINING DESCRIPTION

3.1 Long Hole Stoping

A variant of Cut and Fill stoping is Long Hole Stoping with backfill. This method involves mining out sections of the orebody as “stopes” and then backfilling the void with waste material. This is shown schematically in Figure 4

Figure 4
Schematic of Long Hole Stoping with Backfill



The orebody is divided into “stopes” which are discrete parcels of ore. Typically, the dimension of each stope could be 20m wide, 30m – 40m high and ~50m long, depending on the orebody geometry. The ore within a stope is extracted using long hole drilling and blasting and when all the ore has been recovered the void is filled with waste material. This waste usually contains cemented material which hardens to a solid block which enables the adjoining stopes to be extracted in a likewise way. As shown in the top layer of stopes the centre stope has been mined out and backfilled. The stope to the right is being blasted and loaded out whilst the stope on the left is being drilled.

The main lode at Grandao has a true thickness of 20m however the vertical height is approximately 30m which is very amenable to long hole open stoping (LHOS).

3.2 Decline Access

The surface topography above the lodes is steeply dipping from east to west. The rock above the proposed first development level is somewhat oxidised and probably too unstable to accommodate underground mining. In order to gain initial access to the mineralised zones a horizontal drift will be mined from the surface west of the ore zone to a position on the footwall of the lode. From that point a decline would be mined in the footwall. Because the lode dips at about 30° the access ramp would consist of elongated loops rather than a spiral so as to follow the same dip. The decline would have a gradient of -14% (-1 in 7).

The decline would have a profile of 6.5m by 5.5m which would accommodate thirty tonne all-wheel drive (AWD) trucks.

A ventilation rise would be mined close to the decline to provide ventilation. Crosscuts would be mined off the decline to intersect the rise

3.3 Orebody Access

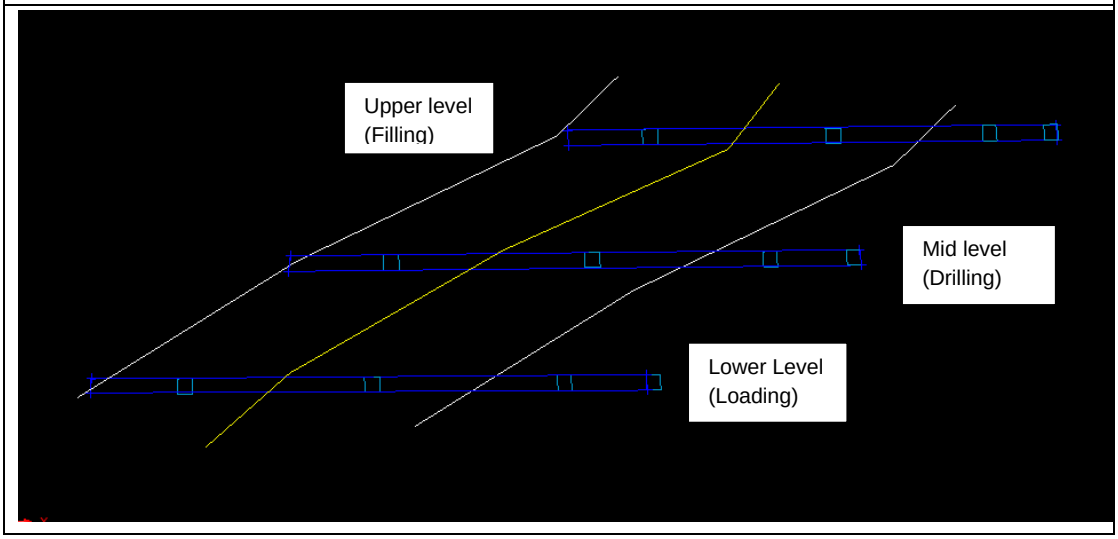
Horizontal crosscuts would be mined from the decline at 30m vertical intervals to intersect the orebody footwall. A horizontal foot wall drive would be mined to parallel the orebody footwall approximately ten metres away from the contact over its entire strike length. The purpose of this drive is initially to provided permanent access to extract the ore for the level above and subsequently to emplace the waste backfill.

Crosscuts would be mined from the footwall drive through the orebody at 50m horizontal intervals to determine the footwall and hanging wall positions. An ore drive would then be mined in the centre of the orebody along its entire length. These ore drives will provide access to mine out the stope above. Where the width of the orebody exceeds thirty metres it is necessary to split the stope and mine it as two separate entities.

The system of drives and crosscuts is repeated at the mid height of the stope and the base of the stope. The ore drive which is at the mid height of the stope becomes the drilling drive for the long holes and the drives at the bottom are used to extract the broken ore. A cross section of the drives and crosscuts is shown in Figure 5.

Figure 5

Section through Orebody Development



The outline of the orebody is represented by the white line. The dark blue lines represent the crosscuts and the light blue squares the drives. The yellow line is the median line of the ore body.

Due to the shallow dip of the orebody there is a risk that blasted ore can sit on the footwall and not gravity feed down to the draw-point level. In this event the method would have to be modified to ensure maximum recovery.

3.4 Drilling and Blasting

Drilling of the long blast holes would be carried out by specialist drill rigs which are computer controlled to ensure accuracy of placement of holes. A drill pattern consists of a series of rings of holes drilled radially from within the drilling drive and extend out to the perimeters of the stope. The number of holes required will depend on the hardness of the rock.

Drill rings are blasted sequentially to slice off the ore along strike however a free face needs to be established before the first ring blast takes place. The free face (slot) is established by drilling a slot rise using the long hole drill. The slot rise consists of concentric circles of holes drill up and down dip around a central “cut” hole. The cut hole is usually a larger diameter than the other holes and is not charged with explosives. The concentric holes around the cut hole are charged and blasted sequentially to create a rise. The rise is further enlarged by drilling and blasting holes which widen the slot out to the full width of the orebody. Once a slot has been blasted to the extents of the orebody the production rings can be blasted usually singly or in pairs.

3.5 Ore Load Out

The blasted ore falls to the bottom loading level of the stope. A front-end loader is used to recover the ore and it is loaded directly into a truck or stockpiled into a “cuddy” for later removal. The loaders will be capable of being operated remotely as it will be necessary for the loader to enter the open stope.

3.6 Haulage

Ore from the stopes will be loaded into thirty tonne capacity trucks and hauled to the ROM pad.

Waste derived from the development activities will be stockpiled underground where possible or stockpiled on the surface for later reclamation. It is proposed that a portion, if not all, of the waste will be backfilled underground.

3.7 Backfilling

To maintain ground stability and maximise ore recovery it will be necessary to fill the stoped-out voids with a cemented fill. This supports the hanging wall and provides a solid roof for work to continue underneath depleted stopes. It is envisaged that the fill will comprise a mixture of cement, tailings and waste rock. This will need to be mixed in a batching plant and delivered underground as a backload by the haulage trucks. Some infrastructure will need to be built to deslime the tailings, crush the rock to a suitable size and mix the cement to the required specifications. It is anticipated that backfilling will take place concurrently with ore production using different stopes.

3.8 Productivity estimation

A typical stope would be 50m long, 30m wide and 30m deep. At an average bulk density of 2.65t/m³ each stope is estimated to contain approximately 120,000tonnes of ore.

The proposed mining method is cyclical in nature and the time taken to complete a full cycle of stope drilling, extraction and backfilling will dictate the maximum rate at which ore can be mined in a year. This assumes that decline and horizontal development progresses well in advance of the stoping.

It is estimated that one metre of long hole drilling will produce 4.5 tonnes of ore therefore it will require 26,700m of drilling to drill out one stope. On the basis that one drill rig can drill approximately 250m per day it will take 100 days to complete drilling.

The rate at which ore can be extracted from a stope is dependent on the loader size and the distance it must travel to dump the ore. The strike length of the orebodies is approximately 600m so a loader would have to travel an average of 150m to load into a truck. Assuming a truck is immediately available it would take approximately 4.6 minutes for each cycle. A loader with a seven cubic metre bucket would achieve production of approximately 1,900 tonnes per day (tpd) consequently it would take 63 days to empty a stope.

Once the stope has been emptied it would have to be filled with a mixture of cemented fill. This material would be sourced from waste mining and from surface stockpiles. It is assumed that it would take the same length of time to fill the stope as it did to empty, it being 63 days.

The drilling, loading and filling cycle would take 226 days to complete.

There are between ten and twelve stopes planned for each level and it is proposed to mine on at least two levels at once. Given the number of faces available it should be possible to extract

from three stopes on one level at once. This would give a potential annual production rate of 570,000 tonnes. There are also smaller lodes which will be mined at the same time which could increase the annual production rate up to 750,000tpa and this target will be used for scheduling purposes.

4 POTENTIAL MINING INVENTORY ESTIMATION

The quantum and geometry of the inventory is determined by the distribution of grade through the orebody and the amenability of solids to be economically extracted.

4.1 Cut-Off Grade Estimation

The economic cut-off grade is determined from the various factors such as cost of production and product price. Table 4.1 shows the costs used in this study to estimate the initial cut-off grade.

Table 4.1
Cut Off Grade Estimation

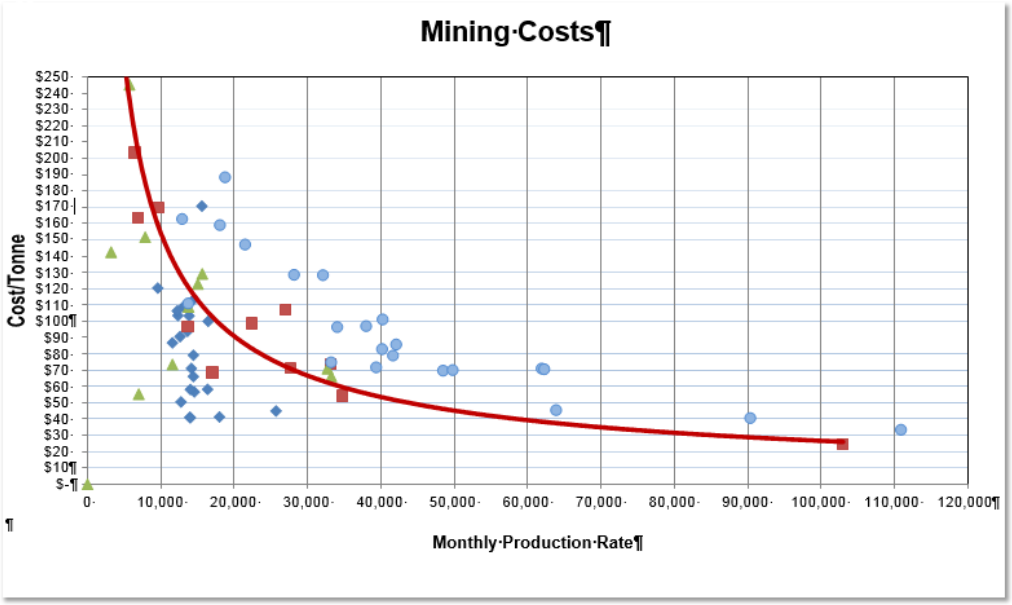
Item	Unit	Value	
Lithium Price	$/t	679	Average price (CIF) over LOM from updated scoping study April 2019
State Royalty (Li)	%	3.0%	No change
Optimisation Li Selling Price	$/t	658.63	Calculated
Metallurgical Recovery Li	%	80.0%	Maintain scoping study assumptions
Realised Value	$/t	526.90	Calculated
Treatment Cost (Li)	$/t ore	18.29	Updated scoping study April 2019 - excludes G&A
Fixed Royalty	$/t ore	0.01	Calculated
Downstream Ore Handling (Li)	$/t ore	8.17	Updated scoping study April 2019 - spodumene transport CIF
By-Products Credits	$/t ore	-9.27	Updated scoping study April 2019 - average annual revenue from pegmatite & bulk byproducts
Admin Cost	$/t ore	2.12	Updated scoping study April 2019 - G&A costs
Tech Services	$/t ore	0.83	Updated scoping study April 2019
Grade Control	$/t ore	1.00	Updated scoping study April 2019
Contractor fixed	$/t ore	10.00	Based on current contract rates
Mining Cost	$/t ore	50.00	Estimated as below
Total Processing Cost	$/t ore	**81.14**	
COG	%	0.92	
Mining Dilution	%	**10%**	
Mining recovery	%	**95%**	
In Situ COG	**g/t**	**1.02**	

Most of the costs and factors are derived from the April 2019 update of the scoping study.

The underground mining cost used in estimating the cut-off grade is based on benchmarking of unit mining costs of several similar underground operations. Figure 6 shows the average unit

mining costs for a range of underground mines. It illustrates that there is an inverse relationship between the monthly production rate and the unit cost of production. The proposed monthly production rate for the Mina do Barroso project is 62,500 tonnes for which the graph indicates a unit mining cost of approximately $50per tonne.

Figure 6
Average Unit Mining Costs



4.2 Estimated Inventory

Inferred resources have been included in this estimation on the basis that they will show the potential of the project to extend to depth.

Whilst the orebody outcrops at surface the top of the orebody has oxidised to a variable extent down to the 490m Rl. It is unlikely that this portion can be mined safely by underground techniques so it is proposed to leave it intact and recover it by open pit methods.

Whilst the resource model bottoms out at approximately 290m Rl there appears to be insufficient tonnage to support extraction to that level consequently the base of extraction has been set at the 310Rl.

Using the vertical limits explained above, the 1.0% li2O cut-off grade and only JORC resources the undiluted inventory is estimated to be 7.6Mt at 1.24% Li2O. Applying mining dilution and recovery factors of 10% and 90% respectively yields an inventory of 7.5Mt at 1.13% Li2O.

5 PROJECT COST ESTIMATION

5.1 Capital Costs

The major cost items in underground mining, assuming contractor mining are as follows:

5.1.1 Mobilisation and Establishment

This is the cost incurred in mobilising all the required buildings, equipment and consumables to site. The mining would be undertaken by a contractor who would establish a set of offices for the staff and a workshop for equipment maintenance.

For this study a value of $2m has been allocated.

5.1.2 Infrastructure

The contractor will supply ventilation fans for the surface and underground and the cost for these will be included in the fixed monthly charge. The Company is not required to supply any infrastructure except a connection to the mine power grid.

5.1.3 Portal Excavation

Access into the workings will be via a tunnel mined into the hillside. In order to provide a safe entry and exit point the ground around the entrance needs to be excavated out and the entrance supported, usually with steel sets. Because the topography is very steeply dipping the portal will only need to be twenty-five metres long at the base. This will give a thickness of six metres of rock over the decline entrance. It will require the excavation of approximately 5,000 bcm of rock. The walls will be supported by split sets and mesh.

5.1.4 Development – Waste

This consists of the decline and inclined and horizontal waste mining. The decline has a total length of 2,138m and will be mined in stages as required.

The inclined waste consists of rises mined between levels to enable the foul air to be drawn out of the mine. These are collectively known as the return air rise (RAR) and connect through to the surface extraction fans. The RAR has a total length of 930m.

The horizontal development connects the decline to the orebodies on the various levels. They consist of the footwall drives and the crosscuts that connect to the orebody footwalls. They comprise a total of 10,945m.

It is necessary to install roof and wall support in the permanent and semi-permanent excavations to prevent rockfalls. Although the ground conditions are presumed to be very stable any unsupported opening is likely to deteriorate after a period of time due to internal rock stresses.

Splits sets will be used to support the roof and approximately 60% of it will be meshed. No allowance has been made to shotcrete any parts of the decline however turn offs will be supported with ten-metre-long grouted cable bolts.

5.1.5 Ore Definition

It will be necessary to more accurately determine the orebody limits and grade by some form of definition drilling. As the orebody is quite shallow this can be done from the surface using reverse circulation drilling. A contingency of $1M has been allowed.

5.1.6 Backfilling

In order to recover the majority of the ore it will be necessary to backfill the stoped voids with cemented fill material.

It is proposed that tailings in conjunction with crushed rock be used to provide the cemented fill material. The tailings would need to be sized to remove the slimes and the rock would need to be crushed and screened to -25mm size to enable it to bind with the tailings correctly. This will require additions to the processing plant in the form of a classifier to de-slime the tailings. A small crushing and screening plant would be required near the portal to treat the waste mined from underground. Additional waste could be sourced from open pit activities if required.

In addition, a batching plant is required to mix the tailings, waste and cement in the correct proportions.

An allowance of $2M is included for the capital cost of the required equipment.

5.1.7 Demobilisation

This cost is incurred when the contractor has completed the work and removes their equipment and facilities form the site. An allowance of $650,000 has been made.

5.1.8 Summary of Capital Costs

The estimated costs are summarised in Table 5.1.

Table 5.1
Estimated Capital Costs

Item	$m
Mobioisation and Establishment	2.0
Development	43.9
Plant and Equipment	2.0
Ore Definition	1.0
Backfill Plant	2.5
Demobilisation	0.7
Total	52.0

The development cost is spread over a period of nine years however the other costs are incurred at the start of the project.

5.2 Operating Costs

Costs associated with waste mining have been capitalised. Costs associated with development in ore and stoping are deemed to be operating.

5.2.1 Development

Operating development costs are incurred in driving crosscuts and strike drives through the orebody and mining the cut-out rises in preparation for stope drilling and blasting. Approximately 12,700m of horizontal ore development is required. This mining produces ore which is trucked to the surface. Approximately 840kt of ore is produced by development mining.

5.2.2 Production

The majority of ore is produced by stoping activities. Approximately 1.5 million linear metres of long holes would be drilled to recover 6.7mt of ore over the 11-year mine life.

5.2.3 Backfilling

A major cost associated with backfilling is the cost of cement. In order for the fill to be strong enough to be self-supporting and hence safe enough to work underneath it will be necessary to use 5% cement in the mix. Currently cement costs approximately $50 per tonne so each tonne of fill will cost $2.50 for the cement. The bulk density of the fill will be approximately 10% less than the bulk density of the in-situ ore consequently approximately six million tonnes of fill, at a cost of $15M will be required.

5.2.4 Owners Team

The owner's team will provide supervision and technical support. The composition is shown in Table 5.2.

Table 5.2
Owners Team

Position	Annual Salary $	No Off	Annual Cost $
Mine Manager	180,000	1	180,000
UG Foreman	120,000	1	120,000
Surveyor	96,000	2	192,000
Survey Assistants	60,0002	2	120,000
Geologists	96,000	3	288,000
Geologist Assistants	60,000	3	180,000
Total		12	1,080,000

5.2.5 Summary of Operating Costs

The quantities and cost of operating activities are shown in Table 5.3.

Table 5.3 Estimated Operating Costs			
Item	Unit	Quantity	Cost $m
Development	Km	12.7	235.0
Production	Mt	6.7	100.3
Backfill	Mt	6.0	15.0
Total			350.2

5.3 Estimated Capital and Operating Costs

The total Estimated Capital and Operating costs are shown in Table 5.41

Table 5.41 Estimated Capital and Operating Costs		
Item	$M	Unit cost $/t ore
Capital	52.0	6.90
Operating	350.2	46.70
Total	402.2	53.60

6 CASHFLOW ANALYSIS

An annual cashflow model has been prepared using the costs and productivities estimated in this study and those used in the optimisation.

6.1 Revenue

There is a gross state royalty of 3% payable for all shipped spodumene. The price of spodumene (6% Li2O) used in this study is $679/t for which $20.37 is subtracted for. the royalty. The estimated metallurgical recovery is 80% which means that 20% of the lithium that is fed into the plant passes through to the tailings dam. In effect the company receives $527 for each tonne of spodumene that is fed to the processing plant. This estimation is shown in Table 6.1.

Table 6.1 Revenue Estimation		
Item	Unit	Value
Spodumene Price (6%)	$	679
State Royalty	%	3
Metallurgical Recovery	%	80
Realised Value	$/t Spodumene	526.90

6.2 Non-Mining Costs

Cost factors used in the cashflow estimation are shown in Table 6.2. These were derived from the April 2019 cashflow model.

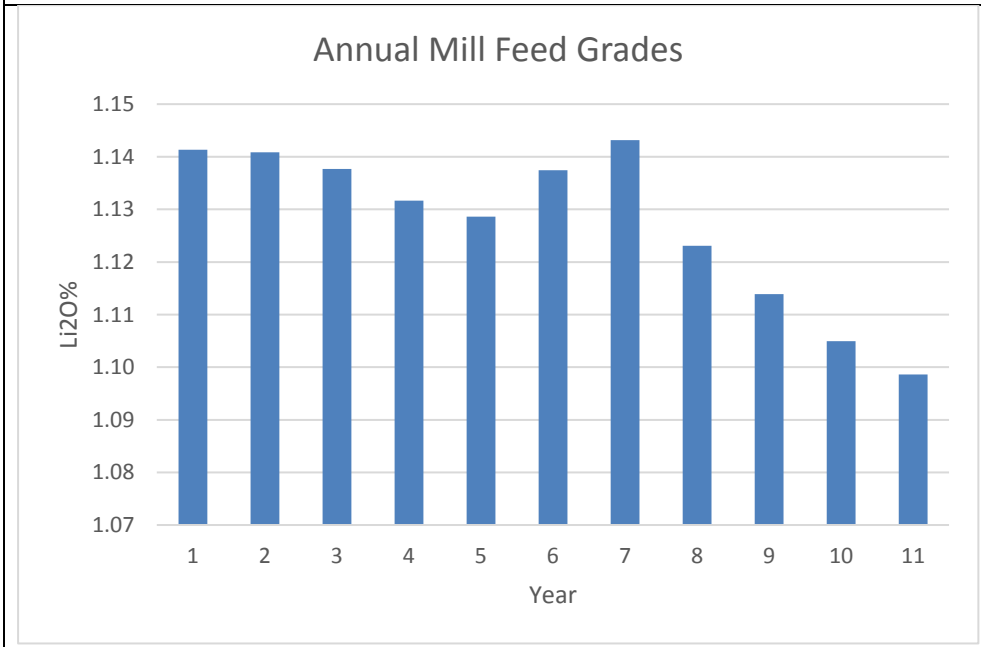
Table 6.2
Non-Mining Cost Factor Estimation

Item	Value $/t
Treatment Cost	18.29
Fixed Royalty	0.01
Downstream Product Handling	8.17
Administration Cost	4.24
Total	30.71

6.3 Project Cashflow Estimation

Annual processing throughput was set at a consistent 750Kt. As shown in Figure 7 the grade of ore varies only slightly on an annual basis although it is probable that the grade could fluctuate by -15% and +40%. The annual grades are shown in Figure 7.

Figure 7
Estimated Annual Mill Feed Grades Li2O%



The revenue and cost factors were used to estimate the potential cashflow (Table 6.3) resulting from the underground mining of the Grandao deposit.

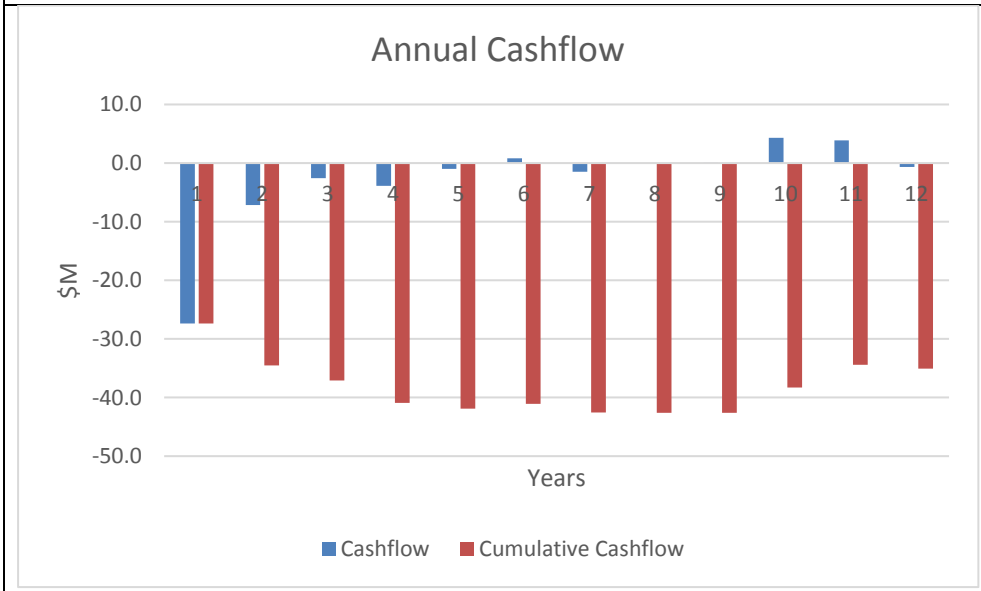
Table 6.3
Project Cashflow Estimation

Item	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Treatment Cost (Li)	137.2	1.0	13.6	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	12.9	0.0
Fixed Royalty	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Downstream Ore Handling (Li)	61.3	0.5	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	5.8	0.0
Admin Cost	31.8	0.2	3.1	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.0	0.0
Mining Opex	350.2	17.9	36.4	34.8	35.8	33.0	32.8	33.5	33.4	32.9	30.9	28.9	0.0
Total Operating Costs	580.6	19.6	59.2	57.8	58.8	56.0	55.8	56.6	56.5	55.9	53.9	50.5	0.0
Capital Costs	52.0	16.2	7.4	4.7	4.7	4.4	3.3	5.1	2.8	2.8	0.0	0.0	0.7
Total Costs	632.7	35.8	66.6	62.5	63.5	60.4	59.1	61.7	59.2	58.7	53.9	50.5	0.7
Revenue	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
k't Spodumene recovered	1,127	8	113	114	113	113	114	114	112	111	110	103	0
Value $M	593.6	4.4	59.5	59.9	59.6	59.5	59.9	60.2	59.2	58.7	58.2	54.4	0.0
Cashflow $M	**-35.1**	**-27.4**	**-7.2**	**-2.6**	**-3.9**	**-1.0**	**0.8**	**-1.5**	**-0.1**	**0.0**	**4.3**	**3.9**	**-0.7**

The estimation indicates that mining of the project would result in a negative cashflow of $35.1M.

The annual and cumulative cashflow are shown in Figure 8

Figure 8
Estimated Annual and Cumulative Cashflow ($M)



Appendix A

Scheduled Mining

Decline Development ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	690	690	0	0	0	0	0	0	0	0	0	0	0
475	0	0	0	0	0	0	0	0	0	0	0	0	0
460	138	138	0	0	0	0	0	0	0	0	0	0	0
445	0	0	0	0	0	0	0	0	0	0	0	0	0
430	375	302	73	0	0	0	0	0	0	0	0	0	0
415	0	0	0	0	0	0	0	0	0	0	0	0	0
400	205	0	147	58	0	0	0	0	0	0	0	0	0
385	0	0	0	0	0	0	0	0	0	0	0	0	0
370	260	0	0	102	158	0	0	0	0	0	0	0	0
355	0	0	0	0	0	0	0	0	0	0	0	0	0
340	260	0	0	0	2	160	98	0	0	0	0	0	0
325	0	0	0	0	0	0	0	0	0	0	0	0	0
310	210	0	0	0	0	0	62	148	0	0	0	0	0
Total	2138	1130	220	160	160	160	160	148	0	0	0	0	0

Off Ramp Access ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	15	15	0	0	0	0	0	0	0	0	0	0	0
475	50	50	0	0	0	0	0	0	0	0	0	0	0
460	80	80	0	0	0	0	0	0	0	0	0	0	0
445	90	0	90	0	0	0	0	0	0	0	0	0	0
430	100	0	100	0	0	0	0	0	0	0	0	0	0
415	120	0	0	120	0	0	0	0	0	0	0	0	0
400	140	0	0	0	140	0	0	0	0	0	0	0	0
385	210	0	0	0	210	0	0	0	0	0	0	0	0
370	270	0	0	0	0	270	0	0	0	0	0	0	0
355	270	0	0	0	0	270	0	0	0	0	0	0	0
340	270	0	0	0	0	0	270	0	0	0	0	0	0
325	310	0	0	0	0	0	0	0	310	0	0	0	0
310	340	0	0	0	0	0	0	0	340	0	0	0	0
Total	2265	145	190	120	350	540	270	0	650	0	0	0	0

Footwall Drive ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	425	425	0	0	0	0	0	0	0	0	0	0	0
475	640	640	0	0	0	0	0	0	0	0	0	0	0
460	850	285	565	0	0	0	0	0	0	0	0	0	0
445	870	0	870	0	0	0	0	0	0	0	0	0	0
430	880	0	660	220	0	0	0	0	0	0	0	0	0
415	940	0	0	940	0	0	0	0	0	0	0	0	0
400	990	0	0	0	990	0	0	0	0	0	0	0	0
385	780	0	0	0	70	710	0	0	0	0	0	0	0
370	575	0	0	0	0	15	560	0	0	0	0	0	0
355	660	0	0	0	0	0	10	650	0	0	0	0	0
340	745	0	0	0	0	0	0	745	0	0	0	0	0
325	570	0	0	0	0	0	0	0	300	270	0	0	0
310	400	0	0	0	0	0	0	0	0	400	0	0	0
Total	9325	1350	2095	1160	1060	725	570	1395	300	670	0	0	0

Waste Cross Cut ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	60	60	0	0	0	0	0	0	0	0	0	0	0
475	90	0	90	0	0	0	0	0	0	0	0	0	0
460	120	0	120	0	0	0	0	0	0	0	0	0	0
445	140	0	140	0	0	0	0	0	0	0	0	0	0
430	150	0	80	70	0	0	0	0	0	0	0	0	0
415	160	0	0	160	0	0	0	0	0	0	0	0	0
400	170	0	0	0	170	0	0	0	0	0	0	0	0
385	150	0	0	0	30	120	0	0	0	0	0	0	0
370	130	0	0	0	0	10	120	0	0	0	0	0	0
355	130	0	0	0	0	0	10	120	0	0	0	0	0
340	130	0	0	0	0	0	0	130	0	0	0	0	0
325	110	0	0	0	0	0	0	0	65	45	0	0	0
310	80	0	0	0	0	0	0	0	0	80	0	0	0
Total	1620	60	430	230	200	130	130	250	65	125	0	0	0

Return Air Rise ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	60	60	0	0	0	0	0	0	0	0	0	0	0
475	0	0	0	0	0	0	0	0	0	0	0	0	0
460	100	100	0	0	0	0	0	0	0	0	0	0	0
445	0	0	0	0	0	0	0	0	0	0	0	0	0
430	180	180	0	0	0	0	0	0	0	0	0	0	0
415	0	0	0	0	0	0	0	0	0	0	0	0	0
400	150	0	0	150	0	0	0	0	0	0	0	0	0
385	0	0	0	0	0	0	0	0	0	0	0	0	0
370	70	0	0	0	0	70	0	0	0	0	0	0	0
355	0	0	0	0	0	0	0	0	0	0	0	0	0
340	210	0	0	0	0	0	0	210	0	0	0	0	0
325	0	0	0	0	0	0	0	0	0	0	0	0	0
310	160	0	0	0	0	0	0	0	0	160	0	0	0
Total	930	340	0	150	0	70	0	210	0	160	0	0	0

Ore Drive ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	385	385	0	0	0	0	0	0	0	0	0	0	0
475	530	290	240	0	0	0	0	0	0	0	0	0	0
460	670	0	670	0	0	0	0	0	0	0	0	0	0
445	740	0	210	530	0	0	0	0	0	0	0	0	0
430	800	0	0	330	470	0	0	0	0	0	0	0	0
415	830	0	0	370	460	0	0	0	0	0	0	0	0
400	860	0	0	0	860	0	0	0	0	0	0	0	0
385	790	0	0	0	0	600	190	0	0	0	0	0	0
370	710	0	0	0	0	0	485	225	0	0	0	0	0
355	700	0	0	0	0	0	0	400	300	0	0	0	0
340	680	0	0	0	0	0	0	0	665	15	0	0	0
325	540	0	0	0	0	0	0	0	0	540	0	0	0
310	390	0	0	0	0	0	0	0	0	70	320	0	0
Total	8625	675	1120	1230	1790	600	675	625	965	625	320	0	0

Ore Cross Cut ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	160	160	0	0	0	0	0	0	0	0	0	0	0
475	280	0	280	0	0	0	0	0	0	0	0	0	0
460	400	0	400	0	0	0	0	0	0	0	0	0	0
445	460	0	230	230	0	0	0	0	0	0	0	0	0
430	520	0	0	260	260	0	0	0	0	0	0	0	0
415	410	0	0	0	410	0	0	0	0	0	0	0	0
400	300	0	0	0	150	150	0	0	0	0	0	0	0
385	300	0	0	0	0	180	120	0	0	0	0	0	0
370	305	0	0	0	0	0	122	183	0	0	0	0	0
355	260	0	0	0	0	0	0	52	208	0	0	0	0
340	220	0	0	0	0	0	0	0	220	0	0	0	0
325	230	0	0	0	0	0	0	0	0	230	0	0	0
310	245	0	0	0	0	0	0	0	0	140	105	0	0
Total	4090	160	910	490	820	330	242	235	428	370	105	0	0
Tonnes	270,963	10,600	60,288	32,463	54,325	21,863	16,033	15,569	28,355	24,513	6,956	0	0
Li2O%	1.13	1.14	1.14	1.13	1.14	1.13	1.12	1.12	1.11	1.10	1.10	0.00	0.00

Slot Rise ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	60	60	0	0	0	0	0	0	0	0	0	0	0
475	60	0	60	0	0	0	0	0	0	0	0	0	0
460	120	0	120	0	0	0	0	0	0	0	0	0	0
445	120	0	60	60	0	0	0	0	0	0	0	0	0
430	150	0	0	80	70	0	0	0	0	0	0	0	0
415	150	0	0	0	150	0	0	0	0	0	0	0	0
400	170	0	0	0	85	85	0	0	0	0	0	0	0
385	170	0	0	0	0	110	60	0	0	0	0	0	0
370	130	0	0	0	0	0	60	70	0	0	0	0	0
355	130	0	0	0	0	0	0	30	100	0	0	0	0
340	130	0	0	0	0	0	0	0	130	0	0	0	0
325	130	0	0	0	0	0	0	0	0	130	0	0	0
310	80	0	0	0	0	0	0	0	0	40	40	0	0
Total	1,600	60	240	140	305	195	120	100	230	170	40	0	0

Stope Drilling ms													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
490	0	0	0	0	0	0	0	0	0	0	0	0	0
475	245,669	0	135,000	110,669	0	0	0	0	0	0	0	0	0
460	0	0	0	0	0	0	0	0	0	0	0	0	0
445	374,225	0	0	30,676	128,242	152,975	62,333	0	0	0	0	0	0
430	0	0	0	0	0	0	0	0	0	0	0	0	0
415	266,315	0	0	0	0	0	90,834	154,006	21,475	0	0	0	0
400	0	0	0	0	0	0	0	0	0	0	0	0	0
385	213,268	0	0	0	0	0	0	0	124,683	88,585	0	0	0
370	0	0	0	0	0	0	0	0	0	0	0	0	0
355	203,922	0	0	0	0	0	0	0	0	63,433	140,489	0	0
340	0	0	0	0	0	0	0	0	0	0	0	0	0
325	176,588	0	0	0	0	0	0	0	0	0	19,921	156,667	0
310	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	1,479,987	0	135,000	141,344	128,242	152,975	153,166	154,006	146,159	152,018	160,410	156,667	0
Tonnes	6,659,940	0	607,500	636,050	577,088	688,388	689,249	693,025	657,714	684,081	721,844	705,003	0
Li2O%	1.13	0.00	1.14	1.14	1.13	1.13	1.14	1.15	1.13	1.12	1.11	1.10	0.00

Appendix B

Annualised Costs

Annual Costs													
Level	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Portal	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Decline	6.3	3.3	0.6	0.5	0.5	0.5	0.5	0.4	0.0	0.0	0.0	0.0	0.0
Cuddies	1.3	0.7	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0
Access	6.6	0.4	0.6	0.3	1.0	1.6	0.8	0.0	1.9	0.0	0.0	0.0	0.0
FW Drives	27.0	3.9	6.1	3.4	3.1	2.1	1.7	4.0	0.9	1.9	0.0	0.0	0.0
FAR	2.6	0.9	0.0	0.4	0.0	0.2	0.0	0.6	0.0	0.4	0.0	0.0	0.0
Waste X/c	4.7	0.2	1.2	0.7	0.6	0.4	0.4	0.7	0.2	0.4	0.0	0.0	0.0
Ore Drives	19.2	1.5	2.5	2.7	4.0	1.3	1.5	1.4	2.1	1.4	0.7	0.0	0.0
Ore X/C	9.0	0.4	2.0	1.1	1.8	0.7	0.5	0.5	0.9	0.8	0.2	0.0	0.0
Slot Rises	0.5	0.0	0.1	0.0	0.1	0.1	0.0	0.0	0.1	0.1	0.0	0.0	0.0
Total	77.2	11.5	13.2	9.2	11.1	6.9	5.4	7.8	6.1	5.0	1.0	0.0	0.0
Stoping	61	0.0	5.6	5.9	5.3	6.3	6.4	6.4	6.1	6.3	6.7	6.5	0.0
Backfill	39	0.0	3.5	3.7	3.4	4.0	4.0	4.0	3.8	4.0	4.2	4.1	0.0
Backfill Produce	15.0	0.0	1.4	1.4	1.3	1.5	1.6	1.6	1.5	1.5	1.6	1.6	0.0
Total	115.3	0.0	10.5	11.0	10.0	11.9	11.9	12.0	11.4	11.8	12.5	12.2	0.0
Mobile Plant	6.4	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.0
Jumbo	10.7	1.3	1.7	1.3	1.5	0.9	0.8	1.2	1.1	0.8	0.2	0.0	0.0
Long Hole	8.2	0.0	0.6	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.0
Trucks	24.0	1.4	2.6	2.3	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.1	0.0
Loaders	18.0	1.3	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.6	0.0
Fuel	26.8	1.7	2.7	2.6	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.4	0.0
Power	4.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.0
Total	98.4	6.7	10.2	9.7	9.7	9.2	9.1	9.4	9.3	9.1	8.4	7.9	0.0
Supervision + Manning	91.3	8.1	8.8	8.5	8.6	8.3	8.3	8.4	8.4	8.3	7.9	7.8	0.0
Owners Team	11.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.0
Grand Total	394.1	27.3	43.8	39.5	40.5	37.4	35.8	38.7	36.2	35.2	30.9	28.9	0.0

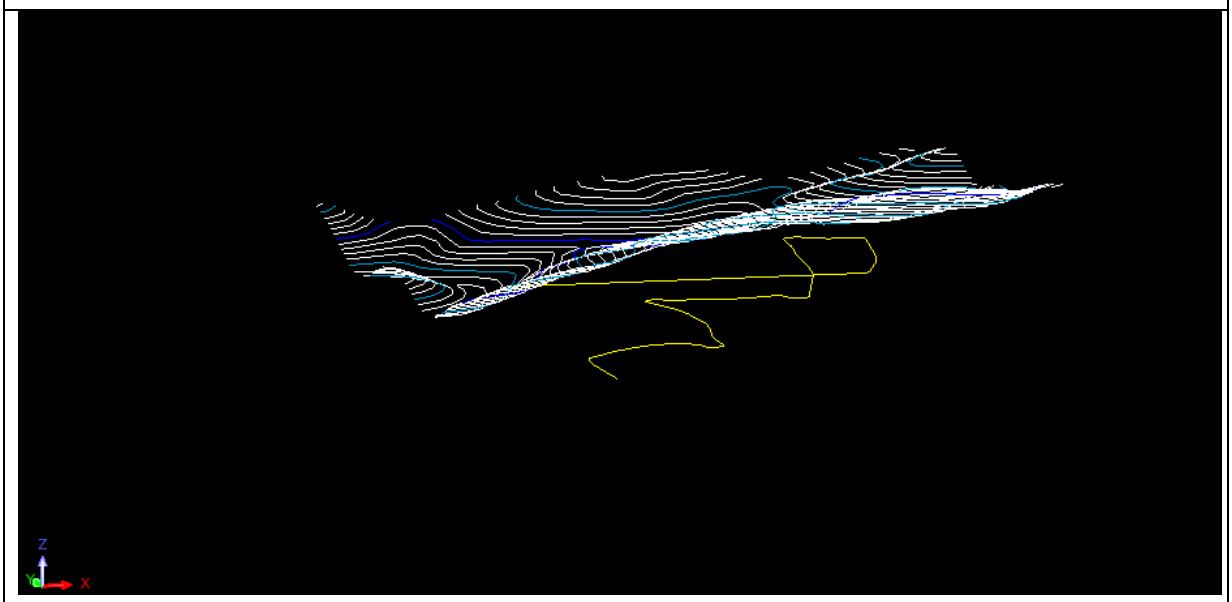
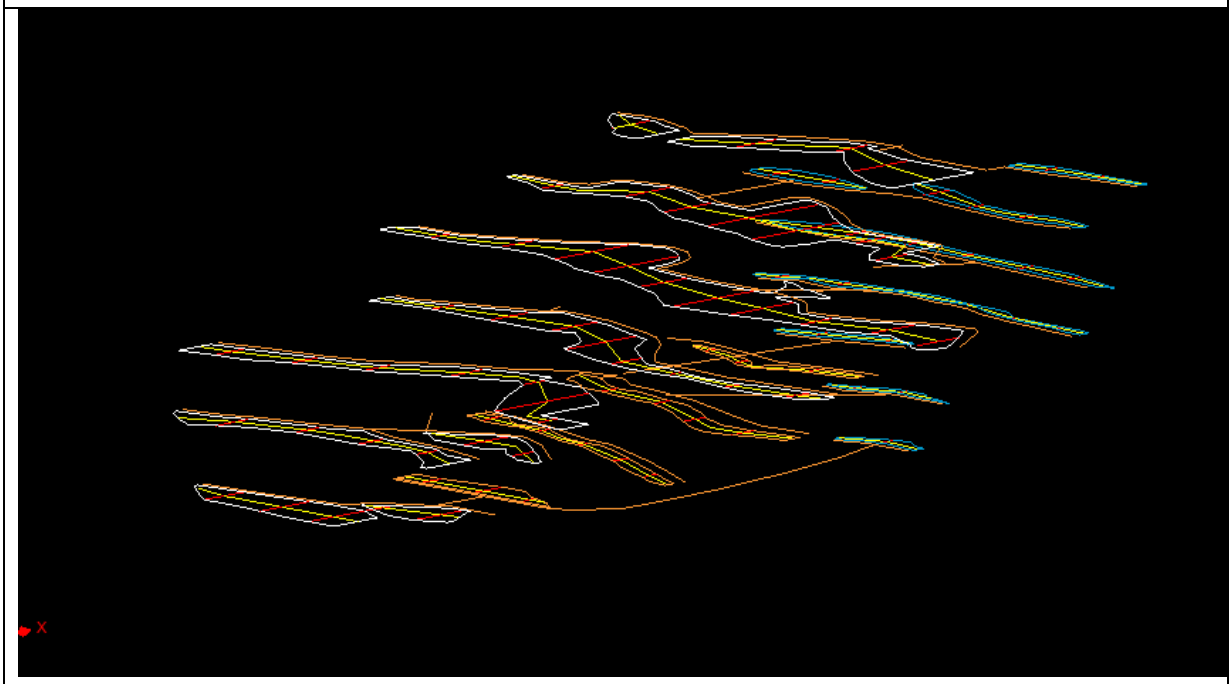
Summary of Capital and Operating Costs													
Item	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Capital Costs													
Establishment + Mobilisation	2.0	2.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Development	43.9	9.4	7.4	4.7	4.7	4.4	3.0	5.1	2.8	2.4	0.0	0.0	0.0
Plant	2.0	2.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Backfill Plant	2.5	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Demobilisation	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.7
Ore Definition	1.0	0.3	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.4	0.0	0.0	0.0
Total	52.0	16.2	7.4	4.7	4.7	4.4	3.3	5.1	2.8	2.8	0.0	0.0	0.7
Operating Costs													
Development	235.0	17.9	25.9	23.8	25.8	21.1	20.8	21.5	22.1	21.0	18.4	16.7	0.0
Production	115.3	0.0	10.5	11.0	10.0	11.9	11.9	12.0	11.4	11.8	12.5	12.2	0.0
Total	350.2	17.9	36.4	34.8	35.8	33.0	32.8	33.5	33.4	32.9	30.9	28.9	0.0
Capital + Operating Costs	**402.3**	**34.1**	**43.8**	**39.5**	**40.5**	**37.4**	**36.1**	**38.7**	**36.2**	**35.6**	**30.9**	**28.9**	**0.7**

Cashflow Estimation													
Item	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Treatment Cost (Li)	137.2	1.0	13.6	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	12.9	0.0
Fixed Royalty	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Downstream Ore Handling (Li)	61.3	0.5	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	5.8	0.0
Admin Cost	31.8	0.2	3.1	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.0	0.0
Mining Opex	350.2	17.9	36.4	34.8	35.8	33.0	32.8	33.5	33.4	32.9	30.9	28.9	0.0
Total Operating Costs	580.6	19.6	59.2	57.8	58.8	56.0	55.8	56.6	56.5	55.9	53.9	50.5	0.0
Capital Costs	52.0	16.2	7.4	4.7	4.7	4.4	3.3	5.1	2.8	2.8	0.0	0.0	0.7
Total Costs	632.7	35.8	66.6	62.5	63.5	60.4	59.1	61.7	59.2	58.7	53.9	50.5	0.7
Revenue													
k'Tonnes Spodumene recovered	1,126.6	8.4	112.9	113.8	113.2	112.9	113.7	114.3	112.3	111.4	110.5	103.3	0.0
Value $M	593.6	4.4	59.5	59.9	59.6	59.5	59.9	60.2	59.2	58.7	58.2	54.4	0.0
Cashflow	**-35.1**	**-27.4**	**-7.2**	**-2.6**	**-3.9**	**-1.0**	**0.8**	**-1.5**	**-0.1**	**0.0**	**4.3**	**3.9**	**-0.7**

Appendix C

Schematics

Schematic showing the decline with regard to the topography



Schematic showing the stoping development levels