ACHMMACH TIN PROJECT

2018 Definitive Feasibility Study Report

Atlas Tin SAS

July 2018

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1 EXECUTIVE SUMMARY

1.1 Introduction

The Achmmach Tin Project (Achmmach or Project) is located approximately 40 km south west of the city of Meknès in central northern Morocco (Figure 1-1). The Project is owned by Atlas Tin SAS, a joint venture company comprising Kasbah Resources Ltd (75%), Toyota Tsusho Corporation (20%) and Nittetsu Mining Co Limited (5%).

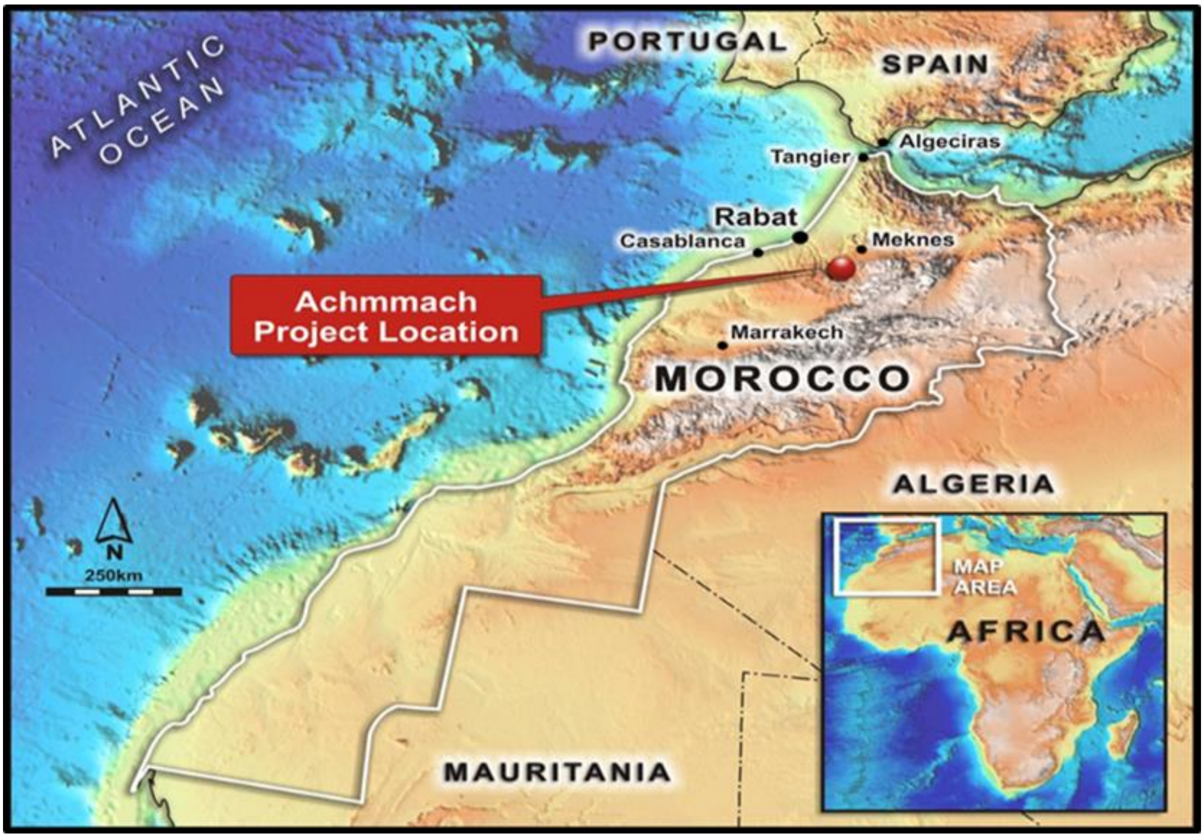


Figure 1-1 Location of the Achmmach Tin Project

This study uses elements of previous studies (2014 DFS (ASX Announcement: 31 March 2014), 2015 EDFS (ASX Announcement: 18 March 2015) and the 2016 SSO (ASX Announcement: 10 August 2016)) and introduces cost and technical enhancements.

1.2 Accessibility, Climate, Local Resources and Physiography

Morocco has well developed national infrastructure including rail, road, sea ports and airports. Access from the capital Rabat to the Project is 150 km east along the A2 expressway to Meknès and then 35 km south along a sealed road to Agourai, and a further 20 km south east along an unsealed rural road to the Project site.

The region has a warm and temperate Mediterranean climate. The temperature varies from 5°C/15°C in winter to 18°C/34°C in summer. The winter months are generally much wetter than the summer months and the average annual rainfall is approximately 700 mm.

The Project is located within rugged terrain of the north-eastern part of the central plateau of the Atlas Mountains. The altitude is 1,085 m above mean sea level (amsl) and the nearby Sidi Addi peak has an altitude 1,230 m amsl. The area is characterised by mountain ranges, valleys and plateaus containing pine forests and oak woodland as well as cleared areas used for agriculture

1.3 Exploration, Geology and Mineralisation

The Achmmach tin deposit is located on the western edge of the El Hajeb province in Northern Morocco. It was discovered by the Moroccan government agency, the Bureau des Recherches et de Participations Minières (BRPM) in 1985. The BRPM explored Achmmach until 1992, completing 14,000 m of diamond core drilling and excavating an exploration shaft to a depth of 80 m. Kasbah commenced exploration at Achmmach in 2007 and has completed a further 105,000 m of diamond core drilling.

The Achmmach tin deposit is hosted within the turbiditic sediments of the Namurian aged Fourhal Formation. The Fourhal Formation consists of deformed, interbedded pelites and psammites of lower carboniferous age. These meta sandstones and shales comprise a tightly-folded sequence of turbidite beds, overprinted by tourmaline alteration within sheared regions and intruded by magmatic sills.

Mineralisation is localised in two subparallel ENE striking lodes named the Meknès and Sidi Addi Trends, separated by a distance of approximately 500 m (Figure 1-2). The largest part of the resource comprises the Meknès Trend. The mineralisation to be mined within the Sidi Addi Trend is referred to as the Western Zone. It is developed within the tourmaline-silica altered metasediments. Tin mineralisation occurs primarily as cassiterite with minor stannite.

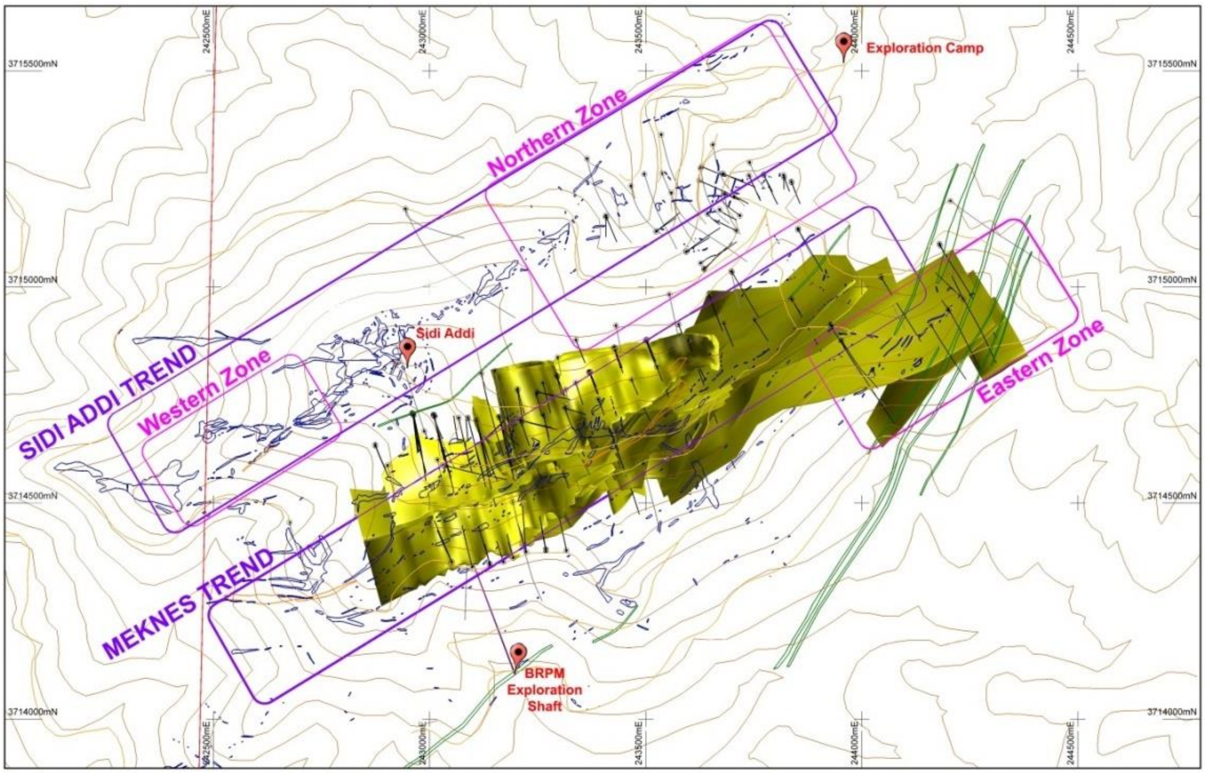


Figure 1-2 Mineralised zones of the Achmmach deposit

1.4 Mineral Resource

Two separate resource estimates have been compiled, for the Meknès Trend and for the Western Zone of the Sidi Addi Trend. The reportable cut-off grade is 0.5% Sn for both the Meknès Trend and Sidi Addi Trend. The Mineral Resource for the Meknès Trend is shown in Table 1.1. The Mineral Resource for the Sidi Addi Trend is shown in Table 1.2. The combined Mineral Resource for the Project is shown in Table 1.3.

Table 1-1 Mineral Resource for the Meknès Trend

Classification	Mtonnes	% Sn	kt Sn
Measured	1.6	1.0	16.1
Indicated	13.0	0.8	107
Inferred	-	-	-
Total	**14.6**	**0.85**	**123.1**

Table 1-2 Mineral Resource for the Sidi Addi Trend

Classification	ktonnes	% Sn	kt Sn
Measured	-	-	-
Indicated	340	1.25	4.2
Inferred	-	-	-
Total	**340**	**1.25**	**4.2**

Table 1-3 Total Mineral Resource for the Achmmach Tin Project

Classification	Mtonnes	% Sn	kt Sn
Measured	1.6	1.0	16.1
Indicated	13.3	0.8	111.2
Inferred	-	-	-
Total	**14.9**	**0.85**	**127.3**

1.5 Ore Reserve Estimate

The Ore Reserve determined as a result of this study is presented below in Table 1.4. The reserve is based on a 0.55% Sn cut-off grade for design. (Calculations have been rounded to the nearest 100,000 t of ore, 0.01% Sn grade and 1,000 t tin metal. Rounding errors may be present.)

Table 1-4 Achmmach Ore Reserve

Zone	Proved			Probable			Total		
	Ore (kt)	**% Sn**	**Tin Metal (t)**	**Ore (kt)**	**% Sn**	**Tin Metal (t)**	**Ore (kt)**	**% Sn**	**Tin Metal (t)**
Meknès Trend	1,100	0.99	11,000	5,600	0.78	44,000	6,700	0.82	55,000
Sidi Addi Trend	-	-	-	300	0.86	3,000	300	0.86	3,000
Total	**1,100**	**0.99**	**11,000**	**5,900**	**0.79**	**47,000**	**7,000**	**0.82**	**58,000**

1.6 Development Design

The Achmmach mine will be accessed via two locations from the surface:

- Central portal boxcut @ 1015 mRL
- Eastern portal boxcut @1085 mRL.

Key features of the design are:

- Each boxcut location has twin portals and declines providing ventilation and escapeway drives parallel to the decline. This eliminates the requirement for raisebored ventilation raises to surface in the Central and Eastern Zones early in the mine plan.
- Three internal declines service the western lodes, eastern lodes and central lodes.
- Independent escapeway drives are developed laterally as a part of each level development and vertically through 1.1 m raises to provide a second means of egress from the production levels once stoping commences.
- The stand-off distance from the decline to the stopes is greater than 25 m, based on geotechnical analysis.
- Diamond drill drives are designed to provide appropriate drilling platforms for grade control drilling programs.

The final mine layout is shown in Figure 1-3.

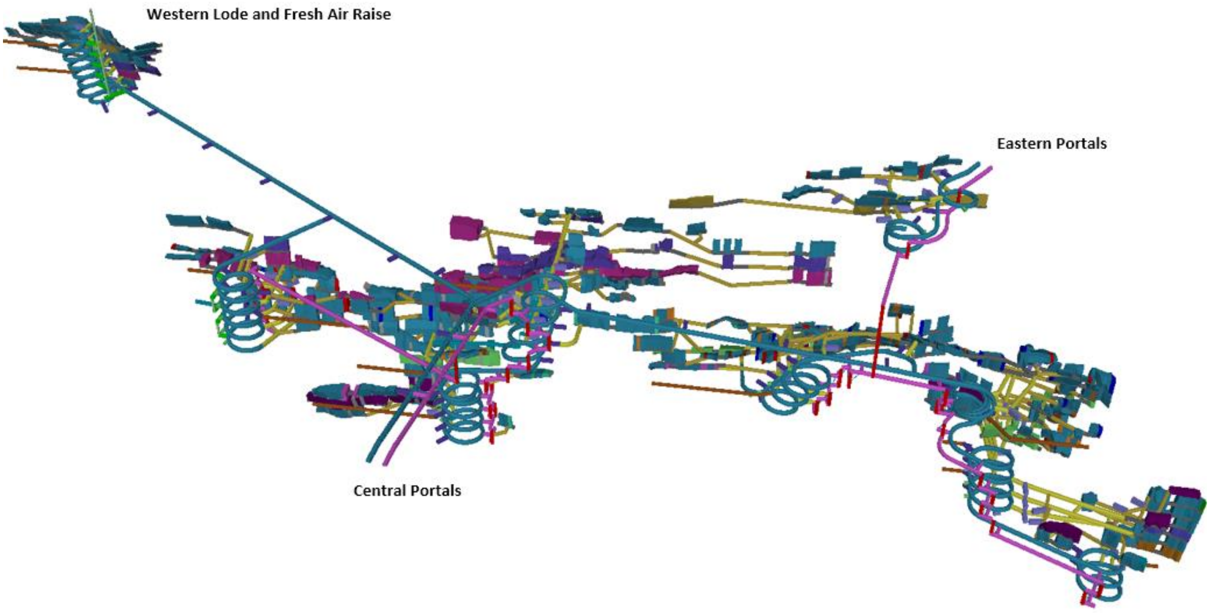


Figure 1-3 Isometric view of development and stope design

1.7 Mining Methods and Mine Layout

The primary mining method to be used for the Achmmach deposit is conventional mechanized longhole stoping. As the geometry and thickness of the mining shapes vary throughout the different lodes, a combination of bottom-up cemented rock fill (CRF) and top-down open stoping methods is planned.

CRF is a simple method of backfilling which involves placement of waste rock mixed with cement slurry into the stope void by a loader from a drive at the top of the stope. Stoping will be carried out by retreating from the extremities to a central access.

The mine design employs CRF in areas of higher grade and greater ore width to minimise metal loss to pillars, with the lower cost open stoping method used in the lower value areas which are developed later in the mine life. There are three zones defined by deposit geometry and proposed mining method; the Central Zone, Eastern Zone and Western Zone (Figure 1-4). For scheduling purposes, the Western Zone has both mining methods applied, with bottom-up CRF above 1015 mRL and top-down open stoping below this point.

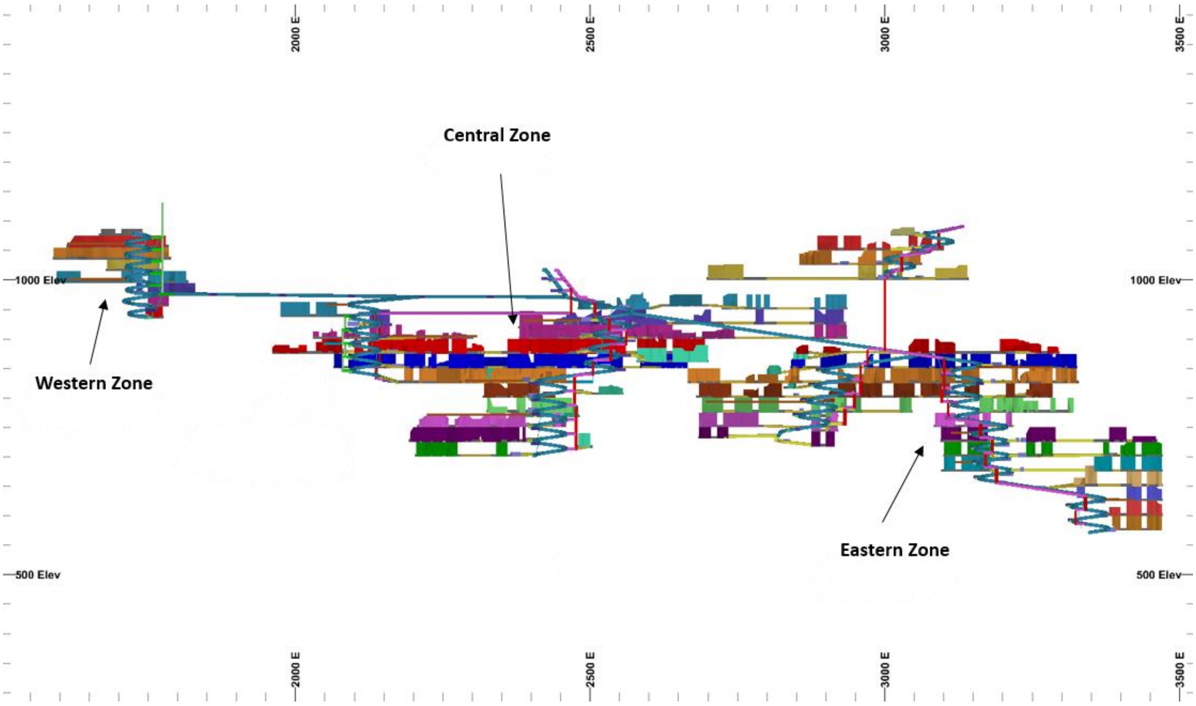


Figure 1-4 Long section showing different mining zones

1.7.1 Central Zone Mining Method

For the generally thick Central Zone (in places up to 20 m in width), a bottom-up mining sequence utilizing CRF will be implemented. A top-down method with no fill, leaving pillars behind for stability was also analysed but this significantly reduced the recovered tonnes due to the required widths of the pillars.

1.7.2 Eastern Zone Mining Method

The Eastern Zone area will be mined using a top-down no-fill method, leaving behind in-situ pillars for stability. The pillar size factors have been extrapolated across the entire Eastern Zone area based on detailed geotechnical design for the Eastern Zone upper which has the lowest strength rock mass.

1.7.3 Western Zone Mining Method

The Western Zone is mined with both the CRF bottom-up method (for levels above the access) and the longhole top-down method (for levels below the access) as described for the Eastern Zone.

1.7.4 Mine Schedule

The mine schedule was created using Datamine enhanced production scheduler. This software integrates detailed data from the resource block model and the mine design directly to the schedule creating a robust output. The process also creates a schedule animation which demonstrates a logical and technically feasible sequence.

The mine production schedule is presented in Table 1-5 and Figure 1-5.

Table 1-5 Summary mine schedule

Physicals	FY 2020	FY 2021	FY 2022	FY 2023	FY 2024	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	Total
Operating Devt (m)		1,863	6,097	4,086	3,528	1,793	1,823	1,724	1,658	1,407	114		24,093
Capital Devt (m)	255	3,821	3,002	3,916	2,415	1,755	1,890	2,024	1,094	47			20,219
Total Devt (m)	255	5,684	9,099	8,002	5,943	3,548	3,713	3,748	2,752	1,454	114		44,312
Total Ore (kt)		213	692	750	726	749	749	752	747	749	670	216	7,013
Sn Grade (%)		0.85	0.83	0.89	0.86	0.88	0.97	0.73	0.75	0.76	0.74	0.72	8.98
Sn Metal (t)		1,810	5,749	6,699	6,242	6,559	7,236	5,512	5,634	5,659	4,992	1,551	57,645

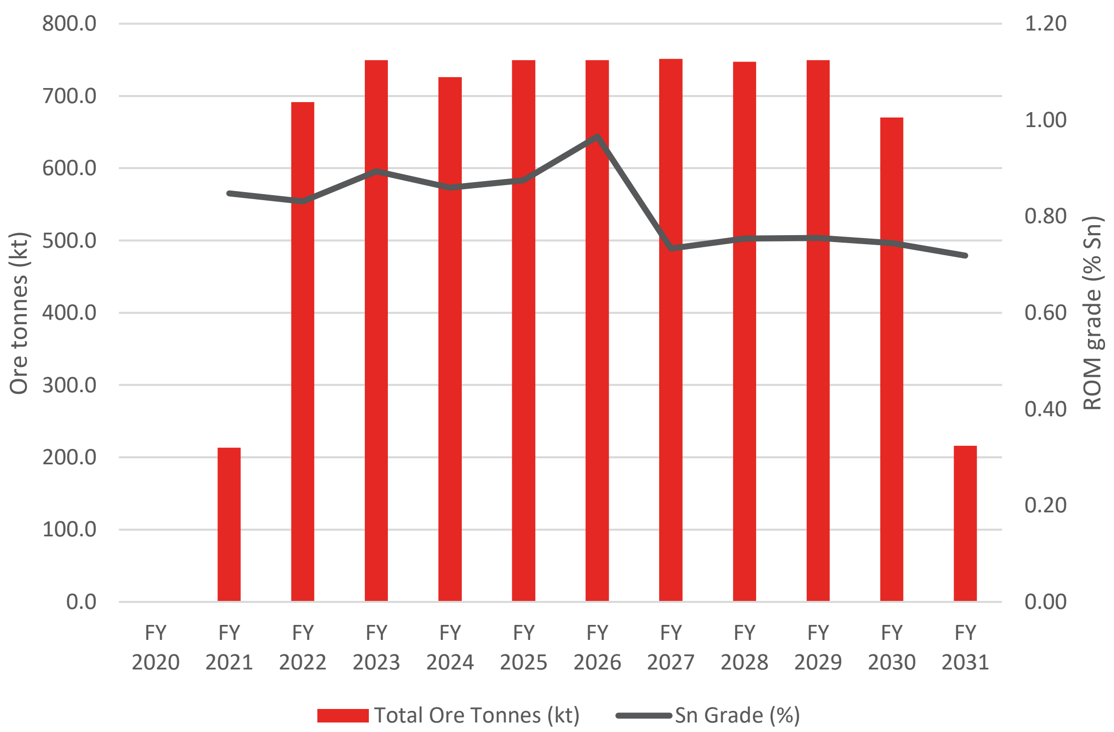


Figure 1-5 Mine plan ore delivery schedule

1.7.5 Mining Sequence

The mining sequence by zone within the overall mine model is illustrated in Figure 1-6.

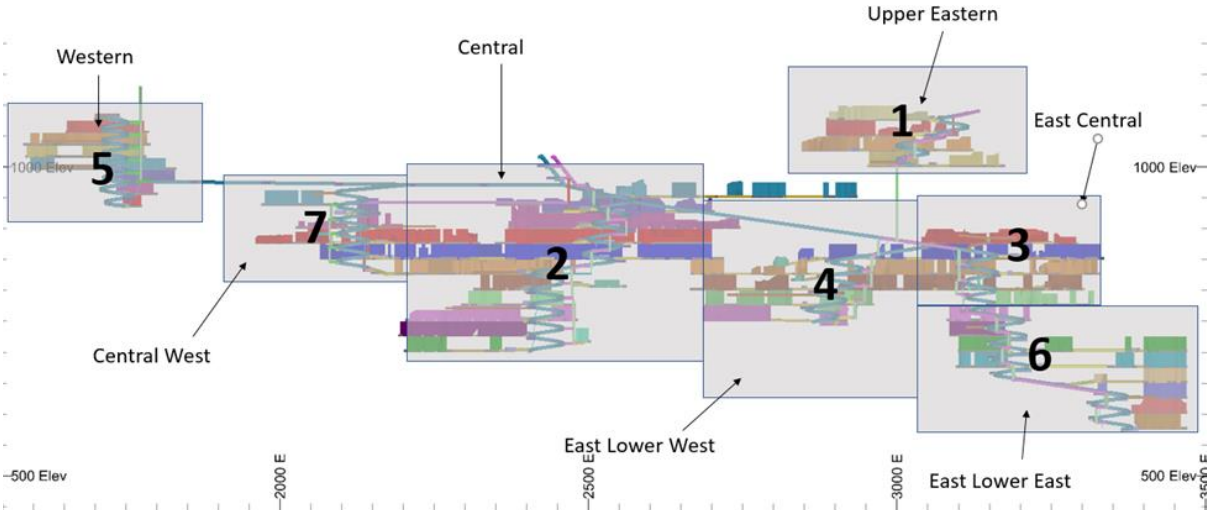


Figure 1-6 Mining sequence (long section looking North)

Development of the central and eastern declines commences simultaneously in the first month. The short decline development into the Eastern Zone will provide early delivery of ore to the run of mine (ROM) pad, allowing mill commissioning to take place while the Central Zone is still under development. The Eastern Zone will provide approximately 40,000 t of ROM ore for 4 to 6 months prior to the delivery of first ore from the larger, more productive Central Zone.

1.8 Metallurgy

The metallurgical behaviour of the Achmmach ore using conventional gravity and flotation processes has been well established by several years of test work by Atlas Tin on representative composite ore samples. The challenge addressed in the test work for this study was to reduce the capital and operating costs of the plant, to provide data on the influence of ore variability in critical design areas, and to seek means to enhance grade/recovery performance.

A pivotal process design change in this study was the introduction of ore sorting. The testwork program evaluated ore sorting as a pre-concentration option for the early elimination of gangue and sub-economic material, thereby potentially:

- upgrading the head grade of tin to the downstream processing plant
- reducing the size (and capital expenditure) of the downstream processing plant
- reducing total consumables, power and water requirements in downstream processing
- increasing the recovery of tin in the gravity and flotation circuit due to the higher grade feed to the downstream processing plant as a result of ore sorting
- improving concentrate grades and thus reducing concentrate shipping and treatment charges
- reducing tailings tonnage, tailings cost and environmental footprint
- using ore sorter rejects for mine backfill and road base applications.

Table 1-6 summarises the predicted metallurgical behaviour based on previous test work and test work conducted on a representative 2018 sample. This shows that the final concentrate grade is 60% Sn at 77% Sn recovery.

Table 1-6 Metallurgical performance predictions from 2018 representative sample

	tph	Ore mass (%)	Grade (%Sn)	Total Sn Rec (%)	OS mass%
Ore feed	94.9	100.0	0.8	100.0	
Fines	22.8	24.0	0.8	24.0	
Ore Sorter feed	72.1	76.0	0.8	76.0	
Accepts product	43.3	45.6	1.4	69.6	60.0
Accepts + fines	66.0	69.6	1.1	93.6	
Rejects	28.8	30.4	0.2	6.4	40.0

Table 1-7 Metallurgical performance predictions from 2018 representative sample

	tph	Ore mass (%)	Grade (%Sn)	Total Sn Rec (%)	Stage Rec (%)
Plant Feed	66.0	69.6	1.1	93.6	
Gravity Concentrate	1.5	1.5	35.0	65.5	70.0
Dressed Gravity conc	0.8	0.8	65.0	62.3	95.0
Coarse Gravity tail	18.5	19.5	0.2	5.0	
Fine Gravity Tail	5.2	5.5	0.5	3.0	
Deslime tail	11.7	12.3	0.6	9.0	
Flotation feed	35.1	37.0	0.4	17.4	
Flotation conc	0.4	0.4	29.0	15.6	90.0
Flotation tail	34.7	36.6	0.0	1.7	
UF Falcon tail	0.2	0.2	3.3	0.7	
UF Falcon concentrate	0.3	0.3	46.0	14.9	95.5
Final Combined Concentrate	**1.0**	**1.1**	**60.2**	**77.2**	**82.4**
Final tail	65.0	68.6	0.2	16.4	17.6
Total recovery from ROM ore					77.2

High pressure grinding rolls (HPGR) testwork also proved to be successful and resulted in the removal of the third crushing stage and the rod mill from previously developed flowsheets in favour of an HPGR.

1.9 Mineral Processing

The process flowsheet is shown in Figure 1-7. Ore delivered by haul truck from underground will be dumped on the ROM pad at a rate of 750 ktpa. The ore will be segregated in grade ranges on the ROM pad to enable blending by the front-end loader into the ROM bin.

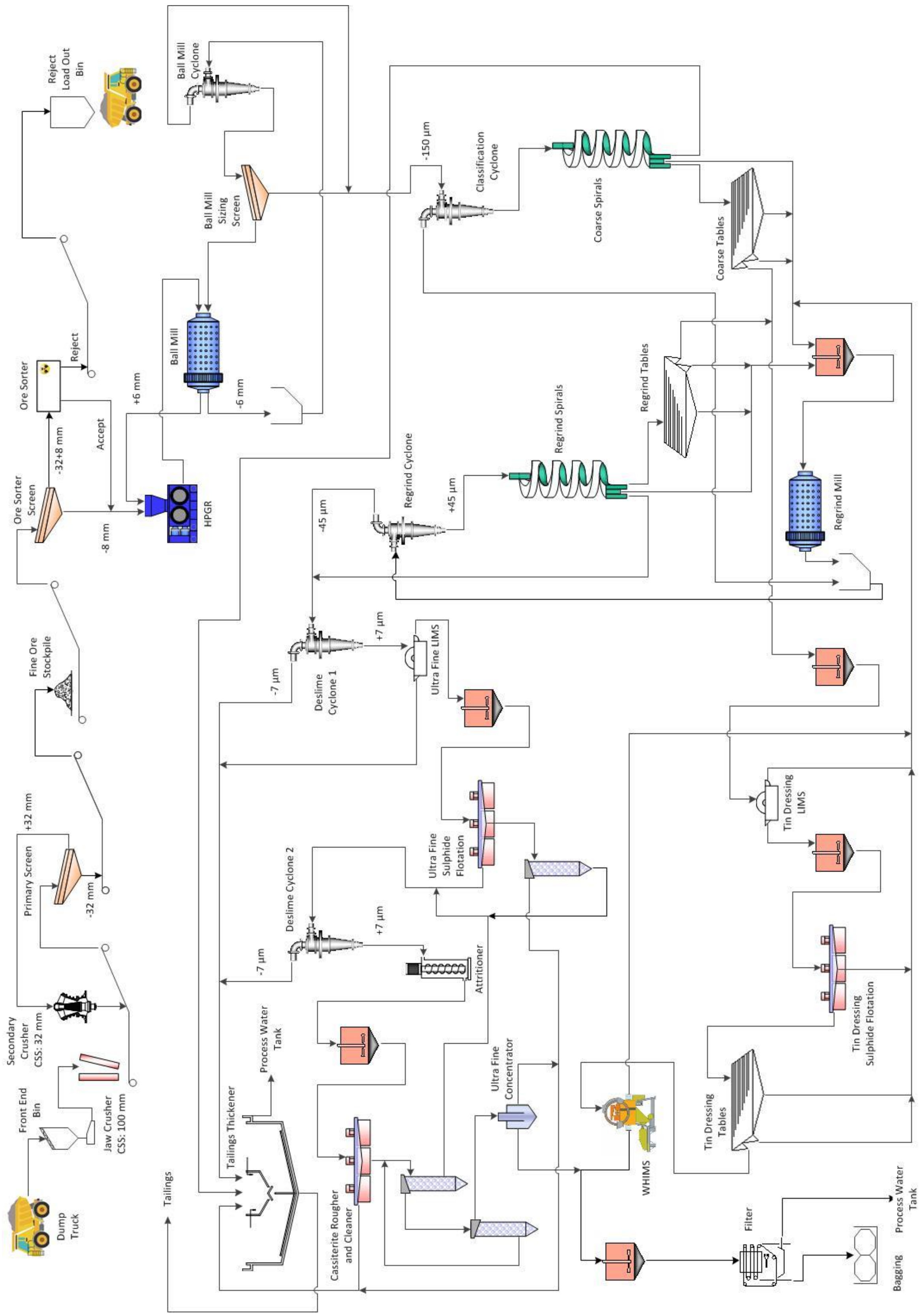


Figure 1-7 Process flowsheet

The two-stage conventional crushing plant will be operated to maintain a full fine ore stockpile (FOS). The secondary crusher will be protected by a metal detector on the feed to the secondary crusher feed bin. The FOS will maintain a nominal feed rate to the fines screen regulated to a weightometer on the screen feed conveyor. The fines screen feed will be diverted to the fines screen undersize conveyor when the ore sorter is stopped. The ore sorter reject rate will be kept nominally to 40% of new feed. This will mean that the positively sorted ore from the ore sorter, termed the accepts, will report to the downstream processing circuits. The ore sorter can be adjusted to compensate for any changes in separation caused by changes in ore grade distribution.

The HPGR is fed with a combination of the ore sorter accepts, the underflow from the fine screen prior to the ore sorter (<8 mm), and a 15% recycle of ball mill scats. The HPGR product is nominally 6 mm P80 at the higher pressure settings. It will be operated as close to a constant tonnage rate as possible to sustain choke feeding and will feed directly into the ball mill.

The ball mill is a short axis grate discharge mill in closed circuit with cyclones and screens. Scats generated by the ball mill at +8 mm size will be conveyed back to the HPGR feed bin. Screen undersize and cyclone overflow, at approximately P_{80} 150 µm, will gravity flow to the coarse gravity cyclone feed pump.

The coarse gravity cyclone will operate at a cut-point of 75 µm, with the cyclone overflow pumped to the regrind (fine gravity) cyclone. The coarse gravity cyclone underflow will feed the coarse spiral and table circuit. The spiral table circuit will operate such that:

- rougher lights are discarded to tails
- rougher middlings feed the scavenger spiral
- rougher and scavenger concentrate feed the cleaner spiral
- scavenger and cleaner middlings plus table tails are directed to regrind
- cleaner tail recycles to the scavenger feed
- cleaner spiral concentrate is directed to tables
- table concentrate will advance to gravity dressing.

The regrind mill grinds to a nominal P_{80} 45 µm and feeds the fine gravity cyclone, which cuts at a nominal 38 µm, and the cyclone underflow feeds the fines gravity spirals and table circuits. The cyclone overflow advances to the first deslime cyclone, and the underflow to the fine spiral and table circuit.

Concentrates from the gravity circuits are further upgraded using magnetic separation to remove steel scrap, then sulphide minerals are removed by flotation and the tail is upgraded to +60% Sn by tabling. A small regrind mill is used to liberate composite cassiterite from table tails.

The first deslime cyclone underflow feeds a magnetic separator to capture steel scats and then flows to a sulphide scavenging circuit. The sulphide concentrate is cleaned to make a sulphide concentrate tail and the (un-floated) tails streams from the rougher and cleaner advance to the second stage deslime cyclone. This cyclone removes any residual slimes and creates the high density slurry required for the high intensity attritioning step ahead of cassiterite flotation. Both cyclone overflows from desliming are sent to the tailings thickener.

The cassiterite flotation has rougher and scavenger stages, with provision for the scavenger concentrate to recycle to the second deslime cyclone. The concentrate from the first stage of cleaning is re-cleaned, the re-cleaner tail recycling to the first cleaner. The first cleaner tail recycles to the second deslime cyclone, and both cleaners have froth washing to minimise gangue entrainment.

The re-cleaner concentrate is passed through a three stage Falcon circuit to remove iron and further upgrade the concentrate.

The final concentrate is filtered in a small plate and frame pressure filter directly filled from an agitated concentrate storage tank. The filter cake is bagged and weighed for shipment.

The concentrate specification based on testwork is listed in Table 1-8.

Table 1-8 Concentrate specification

Element/Compound	Specification
Sn	60.00%
Fe	4.01%
Mn	0.02%
WO_3	0.05%
Pb	0.03%
Zn	0.02%
Ni	0.00%
Co	0.00%
Ag	1.92 ppm
Cu	0.00%
As	0.08%
Bi	0.00%
Sb	0.04%
S	1.73%
$ThO_2+U_3O_8$	0.00%
F	0.09%

1.10 Environmental Studies

1.10.1 Baseline Studies

The physical, social, economic, and cultural baseline has been characterized for the Project using primary information gathered in the field, and secondary information gathered from official sources, such as Government records. Field studies and data gathering for the baseline studies were undertaken between 2007 and 2013.

The general environmental context is that of a sparsely populated mountainous area with valleys and plateau features characterised by forest areas of pine and oak and open land which is used for agricultural purposes. The local population is limited to small villages and individual farms, where the main activity is grazing livestock (sheep, goats and cows) and growing cereal crops.

There are no protected areas in the vicinity of the concession. The nearest environmentally protected area is the Ifrane Park, which is approximately 40 km upstream (to southeast) of the project area and beyond the Project's area of influence.

1.10.2 Tailings Storage Facility

Tailings will be produced by the process plant at a rate of approximately 500,000 tonnes per year. The tailings will be pumped to the tailings storage facility (TSF) located close to the processing plant.

The TSF design is an engineered above-ground valley impoundment, which comprises a basin located in an existing valley floor and a downstream earthen embankment that provides for a maximum depth of tailings of 52 metres at the centre of the embankment after ten years of mine operation. The TSF will be equipped with underdrainage collection to maximize water recovery. The main embankment will be constructed of in-situ mudstone and run of mine waste to form the body of the embankment. The starter embankment will be 27.5 m high (measured from the outer toe of the wall to its crest), with a maximum final height of 52 m at closure of the facility.

A contour drain cut above the upstream extremities of the basin will provide for run-off diversion during rainfall events and snow-melt.

Tailings will normally be deposited at a slurry density of 72% w/w. Initial tailings deposition will be carried out via spigots from the starter wall in an easterly direction to create a beach with a 1% fall towards the rear of the facility. This approach will push the supernatant pool from its initial position against the starter wall up-valley and towards the rear of the facility, until it reaches the toe of the water storage facility. During later years, tailings deposition will be carried out from the western wall raises, as well as from the southern and eastern sides of the facility, the latter to reposition the pool closer to the front of the TSF for closure purposes.

1.10.3 Water Storage Facility

The water storage facility (WSF) will provide the bulk of the project water requirements. It will be constructed directly upstream of the proposed TSF using a cross valley, earthen embankment. Based on the water balance, this embankment will be constructed in a single phase, to a final elevation of approximately 970 mRL. This embankment will form a facility which will have the capacity to store approximately 330,000 m^3 of surface water runoff for use in the mine operations. The facility is an integral part of the TSF water management strategy.

1.10.4 Waste Rock Management

Excavated waste rock will be generated during mine development. The rock which does not contain tin will be transported to a compacted stockpile at the entrance to the mine decline. It is predicted that the rock will be mildly acidic. This potential will be mitigated by blending finely crushed limestone into the waste as it is placed on the stockpile. Mine waste generated during the operational period will be used as fill for the building of the tailings storage facility embankment or as backfill in the underground workings. Backfilling of mine workings will be prioritized. Including ore sorter rejects, the mine waste balance indicates there will be approximately 1.8 Mt of waste on surface at the completion of mining. This includes the use of waste rock for construction of the tailings storage facility. The water storage facility will be constructed from surface cut and fill as it is required to be constructed prior to underground development.

1.10.5 Closure Plan

Closure planning has been undertaken to a conceptual level for key infrastructure such as the TSF and will be continually updated throughout the project life. Prior to the start of production activities, a conceptual mine rehabilitation and closure plan (MRCP) will be prepared. The preparation of the detailed MRCP will be an iterative process that will evolve over the life of the Project taking into consideration views and concerns of the local communities and monitoring information. The MRCP will be finalised at least three years prior to the end of the mine's operating life. Once the mine has finished producing, the detailed closure plan will be implemented.

1.10.6 Permitting

The Project Environment and Social Impact Assessment (ESIA) was prepared in compliance with Morocco's environmental regulations, and in particular law 12-03 relative to environmental impact assessment and Decree n° 2-04-564 regarding public involvement.

An ESIA scoping report was prepared in 2011 and presented to the National Committee for Environmental Impact Assessments (CNEIE) on 15 June 2011. The committee accepted the report and issued terms of reference for the environmental impact assessment.

The ESIA was carried out during the period May 2011 to June 2013. A draft Final ESIA was prepared as part of the 2012 PFS and submitted to the CNEIE in September 2012. An updated version of the ESIA was prepared as part of the 2014 DFS. This included the findings of the social baseline survey that was carried out in April 2013. The document integrates the answers to comments raised by the CNEIE after its initial review in October 2013. The ESIA and environmental and social management and monitoring plan (ESMMP) for the Project were accepted by the Moroccan Ministry of Environment in December 2014 and will expire in December 2019 if there has been no commencement of the Project.

The ESIA was prepared to comply with the International Finance Corporation (IFC) Performance Standards on Environmental and Social sustainability (1st January 2012). The Project is a greenfield mining project and therefore is considered as a Category A project under these standards. The key performance standards that formed the basis of the ESIA are summarized as follows:

- performance standards
- social and environmental assessment and management system
- labour and working conditions
- resource efficiency and pollution prevention
- community health, safety and security
- land acquisition and involuntary resettlement
- biodiversity conservation and sustainable natural resource management
- indigenous peoples
- cultural heritage.

The majority of permits required to support construction and operations have been secured. Remaining permits require detailed design which will commence following the owners approval to continue work following the completion of the feasibility study.

1.11 Social Considerations

The Project is located within the administrative region of Meknès-Tafilalet, in the Caidad of Jahjouh, Cercle of Agourai, El Hajeb Province. The project area is situated in three districts, namely: Ait Ouikhalfen, Jahjouh and Ras Ijerri, which are located inside the Agourai circle. There are 50 households representing an estimated 300 people located within the concession area, of which seven households will need to be resettled prior to the project commencement. Resettlement will be undertaken jointly between the company and the Ministry of the Interior and will align with Moroccan law and IFC Performance Standard 5.

There are no archaeological or formal cultural sites known in the Project area or in the vicinity. There are some spiritual sites within the concession area, such as the Mount Sidi Addi. These sites will not be impacted by the Project.

Local stakeholder consultation has indicated a positive perception of the Project. The Project is seen as an opportunity for local development through job creation and poverty reduction.

To maintain its social licence, Atlas Tin management will develop and implement a social management plan and resettlement action plan. These documents will align with local law and practice and IFC performance standards.

Atlas Tin management has commenced discussions with the local governor’s office regarding establishment of a committee to manage key community, social management and resettlement matters. The local governor’s office will coordinate the committee.

1.12 Market Studies and Contracts

1.12.1 Market Studies

The Project will produce a high-quality 60% Sn tin concentrate. Its quality combined with forecast reductions in global tin production is expected to see the Achmmach tin concentrate highly sought after in international markets.

Three forecast price scenarios are provided as shown in Figure 1-8.

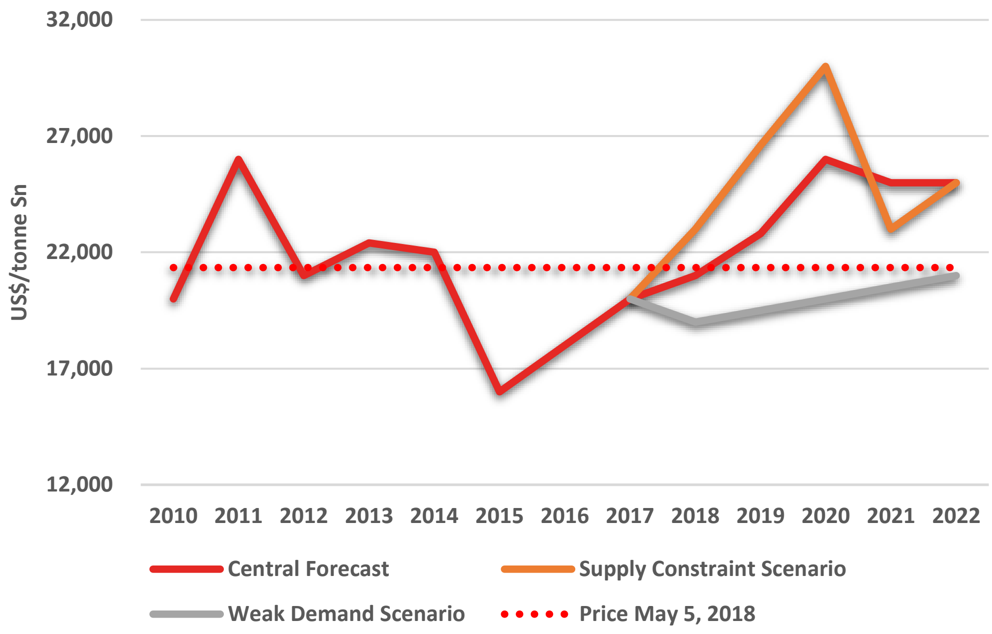


Figure 1-8 Forecast prices for tin (Source: ITA)

Atlas Tin has not entered into any offtake agreement for the sale of its tin concentrate. The joint venture partners are entitled to market 25% of the tin concentrate at market competitive terms. The ability to negotiate offtake and sales agreements for tin concentrate through a positive commodity price cycle provides greater flexibility in negotiations. Atlas Tin has obtained smelter terms from several leading international tin smelting and trading groups as a part of the feasibility study. The tin market has been in deficit in recent years and is expected to remain so in the near term. Ongoing market deficits will not be sustainable going forward and as a result prices are forecast to increase to correct the market imbalance.

1.12.2 Contracts

To underpin cost estimates Atlas Tin has secured pricings for all key commodities and services both domestically within Morocco and internationally where local supply could not be confirmed. These prices were used to support the detailed cost estimates contained within this feasibility study.

To reduce schedule and cost risk, the mine will be developed and operated by an experienced mining contractor. Atlas Tin sought tenders from several contractors which were evaluated and formed the basis of mining rates and cost model inputs. These rates were reviewed and updated in May 2018 as a part of this feasibility study. The final selection of a mining contractor will be based on pricing, capability and capacity criteria.

An engineering, procurement and construction (EPC) Contractor will be engaged to design and construct the processing plant and associated infrastructure on a turnkey basis. This will include contractual obligations relating to technical performance, time and cost, thus managing project risk.

Atlas Tin will secure offtake contracts with credit worthy counterparts to reduce funding risk and secure a revenue stream.

Atlas Tin is currently negotiating a power supply agreement with the Moroccan government agency for energy, the Office National d'Energie et Eau Potable (ONEE). While initial costs have been received for the project, Atlas Tin has yet to enter into any supply contracts for power supply.

1.13 Risk

A project risk workshop facilitated by MYR Consulting was undertaken in May 2018 to identify, analyse and propose control measures to ensure achievement of project objectives. This process considered key risks in relation to the project phases of development, approvals, execution, commissioning/ramp-up, steady state operations and closure.

Corporate and uncontrollable risks were specifically excluded.

Controls to manage key risks will be incorporated into the project management plan.

A total of 40 material risks were identified and categorised according to likelihood and consequence (in accordance with Board approved risk categorisation methodology).

Taking into consideration the existence of current and planned controls, the five major risks identified in order of decreasing severity were as follows:

- higher than expected ROM variability
- lower than forecast metal tonnes
- delay/poor performance of mining contractor
- serious security, safety or health incident
- loss of support from key stakeholders.

Atlas Tin will develop a risk management plan for construction and operations. The risk management process will be the responsibility of the Environmental Health & Safety (EHS) manager who will report monthly to the General Manager on all aspects of the risk management process.

1.14 Capital and Operating Costs

1.14.1 Project Capital Costs

The initial Project capital costs have been estimated as US$96.4M (real 2018 $) as summarised in Table 1-9.

Table 1-9 Project capital costs

2018 DFS Project Capital Costs	US$M
Mining development	12.1
Tailings storage facility and water storage facility	3.5
Process plant	44.5
Infrastructure	12.0
Engineering, Procurement and Construction (EPC)	7.2
Construction indirect costs	5.4
Sub-total Project Construction Capital	**84.7**
First fill & spares	1.2
Contingency	10.5
Total Project Capital Costs	**96.4**

1.14.2 Sustaining and Mine Closure Capital

Sustaining and replacement capital costs for mining and processing have been estimated at US$69.2 M over the life of mine. These costs largely relate to the ongoing mine development which is timed to be completed on an as required basis to optimise project cashflow. Approximately three quarters of the capital mine development is categorised as sustaining.

The mine closure cost has been estimated to be US$3.1 million. Salvage values from dismantling and selling of the process plant has been estimated at US$2.7 million which will offset the majority of the estimated closure costs.

1.14.3 Operating Costs

The total operating costs are summarised in Table 1-10 below.

Table 1-10 Project operating costs

Category	US$ Millions	US$/t Sn
Mining	216.6	4,866
Processing	109.8	2,466
Administration	36.7	823
Concentrate transport & treatment	45.4	1,021
C1 cash costs	**408.5**	**9,176**
Depreciation & amortisation	165.4	3,815
C2 costs	**573.9**	**12,991**
Royalties	25.2	566
Corporate costs	6.1	138
C3 costs	**605.2**	**13,695**
Sustaining capital	69.2	1,554
All in sustaining cash costs (AISC)	**509.0**	**11,435**

C1 cash costs consist of mining, processing, administration and concentrate transport & treatment. AISC consist of C1 cash costs, royalties, corporate costs and sustaining capital.

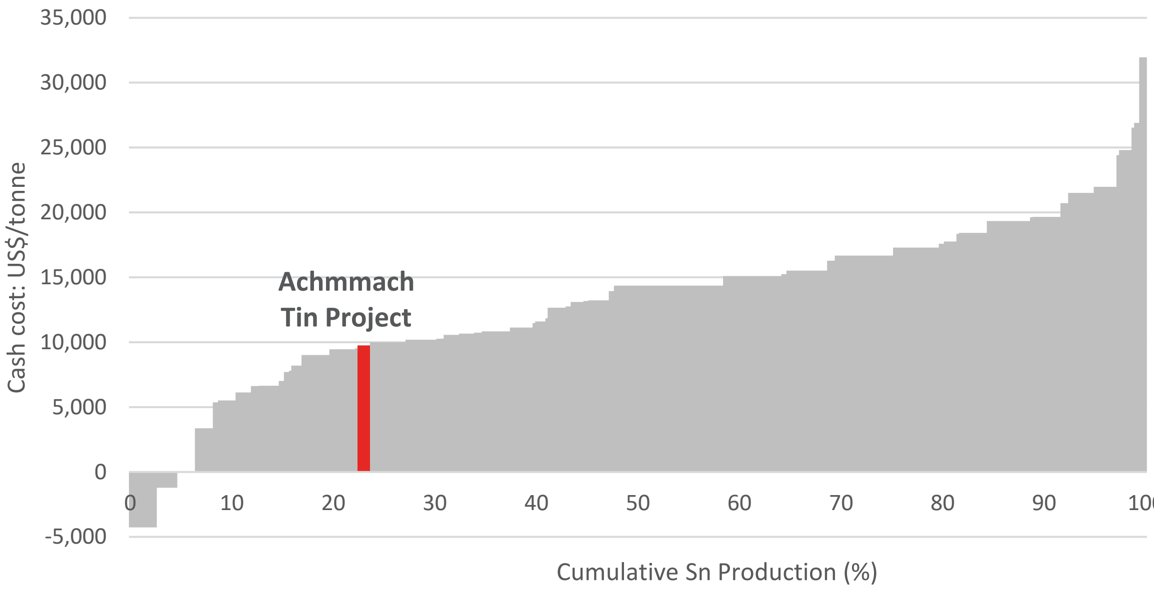


Figure 1-9 Tin producer cost curve (source ITA).

The cost curve shown in Figure 1-9 is based on tin producer information published by the International Tin Association (ITA) and reflects the estimated cash costs of tin producers in 2021. ITA has defined cash costs as the cost of mining, processing, smelting, marketing, transport and any royalties or taxes, net of any by-product revenue.

As shown in the cost curve, the Project will be an upper first quartile/lower second quartile tin cost producer.

1.15 Economic Analysis

The Project is a joint venture. All the numbers presented in this financial evaluation section of the report are based upon 100% ownership.

Furthermore, the Project is evaluated on a 100% equity basis only and excludes any financial leveraging effects (i.e. ungeared), as well as any interest expense items that could impact taxable income and/or provide interest deduction tax shields.

The analysis is based on a processing throughput of 750 ktpa using the Ore Reserves defined by this feasibility study.

The objective of the economic analysis is to demonstrate the economic viability of the Project, provide support for project financing activities, and enable the shareholders to reach formal decisions to proceed with the detail design and construction phases of the project.

1.15.1 Evaluation Methodology

The financial evaluation has been performed using real cash flows. No escalation of cash inflows or outflows has been applied in determining the project NPV and IRR.

The evaluation is based on after-tax unleveraged, real internal rate of return (IRR) using monthly cashflows using mid period convention.

Exchange rates used in the project cost analysis and evaluation are detailed below in Table 1-11. These exchange rates have been determined based on spot prices dated 2 July 2018.

Table 1-11 Project exchange rates

Currency	Rate of exchange to USD
AUD	0.745
ZAR	0.075
EUR	1.157
MAD	0.105

1.15.2 Base Date

The valuation date for the financial model is 1 April 2019, reflecting when the decision to construct the Project is anticipated to be made.

1.15.3 Revenue

On-mine revenue is derived from the sale of tin concentrates into the international market place. Revenues are based on the value of tin content of the concentrate less unit deductions, smelting and refining and impurity penalty charges. No marketing fees have been reflected in the model as Atlas Tin expects to enter into long term offtake agreements with one or more traders/smelters with FOB shipping terms, where ownership and control ceases once loaded at the port of Casablanca.

1.15.4 Tin Price Forecast

For purposes of financial modelling, a price of US$21,000/t (real 2018$) has been used. This price reflects the recent spot price and has also been determined based on the 105-month historical average plus 15-month LME futures contract.

1.15.5 Marketing and Treatment Charges

Through a market soft-sounding, Atlas Tin gained indicative non-binding offtake terms for its tin concentrate from a number of traders and smelters to established indicative net smelter returns (NSR) and average metal payability primarily for financial model purposes. Concentrate treatment charges have been applied which include penalties and unit deductions for impurities.

1.15.6 Production and Revenue Summary

Construction of the Project is scheduled to take 20 months, before first production in the second half of 2020. A six-month production ramp-up has been scheduled for commissioning of the plant before achieving full design capacity.

1.16 Revenue Inputs and Assumptions

Revenues are calculated in United States dollars based on a metal price of US$21,000/t (real 2018$). Penalties and unit deduction charges are deducted in calculating the on-mine revenue.

The financial model assumes that 90% of the revenues are received in the month following production, with the balance received 3-months later.

1.17 Financial Summary

Table 1-12 is a summary of the financial analysis of the project.

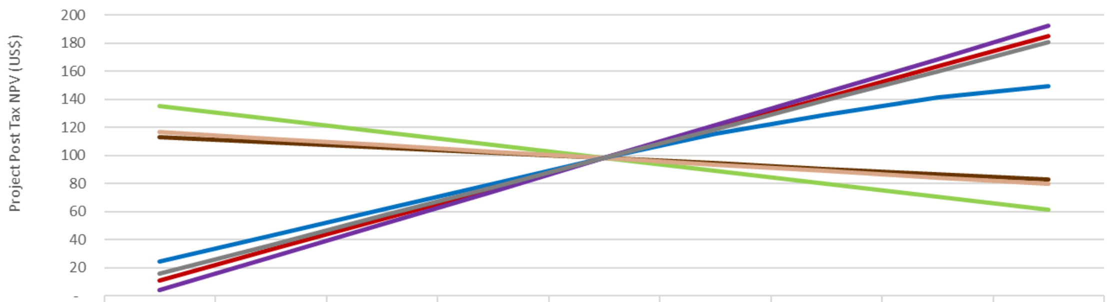
Table 1-12 Financial summary (US$)

Parameters	
Sn Price	US$21,000/t
Discount rate (real)	8%
NPV, post tax[1]	$98.1M
IRR, post tax[1]	23%
Capital costs	$96.4M
C1 cash costs	**$9,176/t**
C3 costs	**$13,695/t**
Average AISC	**$11,435/t**
Operating cash flow	$403M
Free cash flow	$267M
Turnover	$815M
EBITDA	$444M

[1] Project NPV and IRR based on a post-tax discount rate of 8% and Moroccan Corporate Income Tax of 17.5%

1.18 Sensitivity

The Project economics are most sensitive to changes in the tin price, grade and recovery. The sensitivity of the NPV to changes in major cost and revenue drivers is shown in Figure 1-10 below.

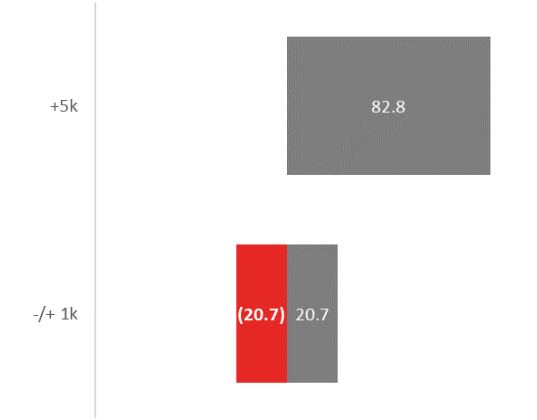


	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Sn Price	11	33	55	76	98	120	142	163	185
Capital Costs	113	109	106	102	98	94	91	87	83
Operating Costs	135	126	117	107	98	89	80	70	61
Exchange Rate	117	112	107	103	98	93	89	84	80
Throughput	24	43	61	80	98	115	129	141	150
Grade	4	27	51	75	98	122	145	169	192
Recovery	15	36	57	77	98	119	139	160	181

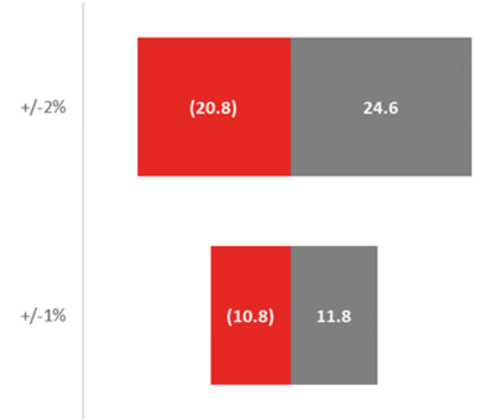
Figure 1-10 Project NPV sensitivity

Detailed NPV and IRR sensitivity analysis was performed on the major cost and revenue drivers and the NPV analysis is presented for the individual factors in Figure 1-11.

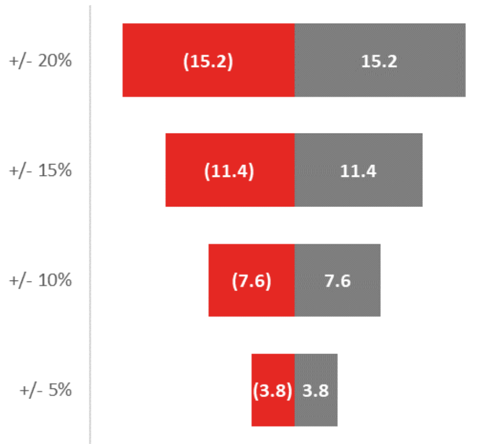
Tin Price Sensitivity



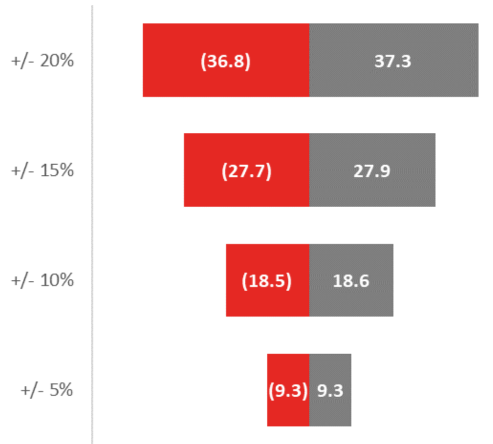
Discount Rate Sensitivity



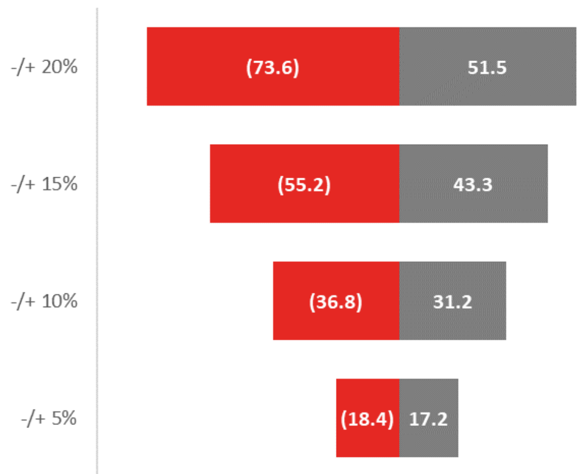
Capital Cost Sensitivity



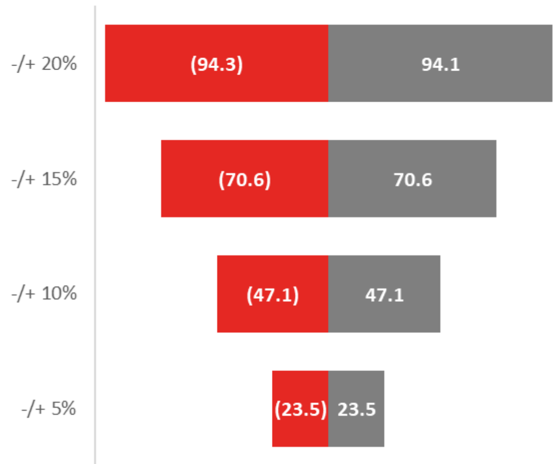
Operating Cost Sensitivity



Throughput Sensitivity



Grade Sensitivity



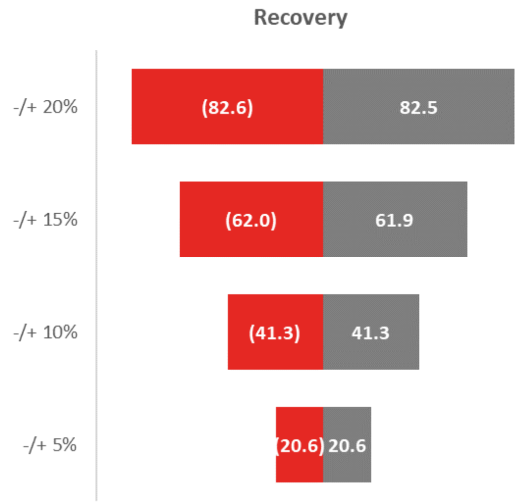


Figure 1-11 Detailed sensitivity analysis

1.19 Mine Life

The operating mine life of the Achmmach Project, based on the assumptions in the definitive feasibility study, is 10 years. This includes the initial production ramp-up but excludes project construction, pre-production and mine closure activities.

Extending the life of the mine would rely on the conversion of measured and indicated resources to reserves and the discovery of additional resources within the current mining permit area. The existing orebodies are open along strike and at depth, providing excellent exploration potential.

1.20 Recommendation

The feasibility study demonstrates that the project can produce a robust positive economic result and it is recommended that the project is progressed to enable a final investment decision which will include, *inter-alia*, gaining committed funding and concluding major commercial contracts such as EPC, mining contractor and offtake arrangements.

1.21 Glossary

Agencies

ABHS	*Agence du bassin hydraulique du Sebou* (Water Basin Agency)
AMDI	*Agence Marocaine de Developpment d'Investissements* (Moroccan Investment and Development Agency)
BRPM	*Bureau de Recherches et de Participations Minières*
CNEIE	National Committee for Environmental Impact Assessments
ICDX	Indonesian Commodities and Derivatives Exchange
IFC	International Finance Corporation
ISO	International Standards Organisation
ITRI	International Tin Research Institute
JORC	Joint Ore Reserves Committee
LME	London Metal Exchange
MEMEE	Ministry of Energy, Mines, Water and the Environment
MoI	Ministry of the Interior
NGO	Non-governmental organisation
OECD	Organisation for Economic Cooperation and Development
ONEE	*Office National de l'Electricité et de l'Eau Potable*
ONEP	*Office National de l'Eau Potable (*National Office of Potable Water (superseded))
ONHYM	*Office National des Hydrocarbures et des Mines*

Companies

AHK	Alfred H Knight
Artelia	*Artelia Eau & Environnement*, Grenoble, France and Rabat, Morocco
BNPP	BNP Paribas
BRL	Burnie Research Laboratory, Burnie Tasmania Australia
Cap	Cap Rurale, Meknès Morocco
Carras	Carras Mining Pty Ltd, Perth Western Australia
CPC	CPC Project Design Pty Ltd, Perth Western Australia
CRU	Commodities Research Unit, London
DRA	DRA Pacific Pty Ltd, Perth Western Australia
FLS	FL Smidth
Galay	Galay BTP Construction, Meknès Morocco
Golder	Golder Associates (UK) Ltd, England
Mining One	Mining One Pty, Melbourne Victoria and Perth, Western Australia
Optimum	Optimum Capital Pty Ltd, Perth Western Australia
Outotec	Outotec Pty Ltd, Perth Western Australia
QG	Quantitative Group, Fremantle, Western Australia
SAMINE	*Société Anonyme d'Entreprises Minières*
SGS	SGS Lakefield Research Europe
SLON®	SLON Company
TTC	Toyota Tsusho Corporation

Currencies

AUD	Australian Dollar
CAD	Canadian Dollar
CNY	Chinese Yuan
EUR	Euro
GBP	Great Britain Pound
MAD	Moroccan Dirham
SEK	Swedish Krona
USD/US$	United States Dollar
ZAR	South African Rand

Chemicals and Reagents

Ag	Silver
Al_2O_3	Aluminium Oxide
As	Arsenic
Co	Cobalt
Cu	Copper
F	Fluorine
Fe	Iron
Ga	Gallium
H_2SO_4	Sulphuric acid
In	Indium
MIBC	Methyl Isobutyl Carbinol
Mn	Manganese
Nb	Niobium
Ni	Nickel
Oxidisable S	Oxidisable sulphur
PAX	Potassium Amyl Xanthate
Pb	Lead
S	Sulphur
Sb	Antimony
SiO_2	Silicon dioxide
Sn	Tin
Ta	Tantalum
Ta	Tantalum
WO_3	Tungsten trioxide
Zn	Zinc

Metrics

µm	Micrometre
g/L	gram per litre
g/t	gram per tonne
Ga	10^9 years
h/a	hours per year
Ha	Hectare
kg	kilogram

$kg/h/m^2$	kilogram per hour per square metre
kg/m^3	kilogram per cubic metre
kg/t	kilogram per tonne
km	kilometre
kPa	kilopascal
kPag	kilopascal gauge
kt	Thousand tonnes
kV	kilovolt
kW	kilowatt
kWh	kilowatt hour
kWh/t	kWh per tonne
L	Litre
L/s	Litre per second
m	metre
M	Million
m H	metres high
m W	metres wide
m/d	metres per day
m/month	metres per month
m^3/day	cubic meters per day
m^3/h	cubic meters per hour
m^3/kW	cubic meters per kilo watt
m^3/s	cubic meters per second
Ma	10^6 years
Ma	Million years
masl	metres above sea level
mE	metres Easting
mm	millimetre
Mm^3	Million cubic metres
MPa	MegaPascal
mRL	metres reduced level
Mt	Million tonnes
Mth	month
Mtpa	Million tonnes per annum
MVA	Megavolt. Amps
No.	number
RL	Reduced level
t	tonne
t.km	tonne kilometre
t.km/y	tonne kilometre per year
t/d	tonnes per day
t/h	tonnes per hour
$t/h/m^2$	tonnes per hour per square metre
t/y	tonnes per year
tpa	tonnes per annum

V	Volt
w/w	Weight per weight

General Terminology

ANC	Acid Neutralisation Capacity
ANFO	Ammonium nitrate – fuel oil (explosive)
BA	Bench Angle
BH	Bench Height
BMC	Bulk Meknès Composite
BSA	Bench Stack Angle
BSW	Bench Stack Height
Capex	Capital Expenditure
CCL	compacted clay liner
COF	Cyclone overflow
COG	Cut-off grade
Con	Concentrate
CUF	Cyclone underflow
DD	Due diligence
DFS	Definitive Feasibility Study
Distn	Distribution
DMC	Dense Media Cyclone
DMS	Dense Media Separation
E&I	Electrical and instrumentation
ENE	East North East
EPCM	Engineering, Procurement and Construction Management
EPS	Enhanced Production Scheduler
ERI	Electrical Resistivity Imaging
ESE	East South East
ESIA	Environmental and Social Impact Assessment
ESMMP	Environmental and Social Monitoring and Management Plan
E-W	East West
EZS	Eastern Zone Shallows
FEED	Front end engineering and design
FIFO	Fly in Fly Out
GP	General Portland
GTB	Geotechnical Berm
HAZID	Hazard Identification
HAZOP	Hazard and Operability
HDPE	High-density polyethylene
HLS	Heavy liquid separation
HSE	Health, safety & environment
HT	High tension
IP	Induced Polarization
IRA	Inter Ramp Angle
ITH	In-The-Hole-Hammer

KPIs	Key Performance Indicators
LH	Low Heat
LHOS	longhole open stoping
LIMS	Low Intensity Magnetic Separator
LOM	Life of Mine
LTI/SPI	Lost time injury / Serious potential incident
LV	Light Vehicle
MRCP	Mine reclamation and closure plan
MSO	Mine Stope Optimiser
MTI	Medical treatment injury
n/a	not applicable
NAG	Net Acid Generation
NAPP	Net Acid Producing Potential
NE	North East
NNE	North East
NPV	Net present value
NQ and HQ	drill gauges
NSR	net smelter return
OEM	Original equipment manufacturer
OSA	Overall Slope Angle
Oued	River
P&G	Provisional & General
pa	per annum
PDC	Process design criteria
PETN	Pentaerythritol tetranitrate (an explosive)
PFS	Prefeasibility Study
PSD	Particle size distribution
QEMScan	Quantitative electron microscope scan
RoHS	Restriction on Hazardous Substances
ROM	Run of mine
SAG	Semi-autogenous grinding
SBW	Spill Berm Width
Scav	Scavenger
SEM	Scanning Electron Microscope
SFE	Shake flask extraction
SMC	Steve Morell Comminution
SW	South West
TSF	Tailings Management Facility
UCS	Unconfined Compressive Strength
UF	Ultrafine
UTM	Universal transverse Mercator
VAT	Value Added Tax
WBS	Work Breakdown Structure
WHIMS	Wet high intensity magnetic separation
WNW	West North West
WSF	Water Storage Facility

WSW	West South West
WZS	Western Zone Shallows
XRF	X-Ray fluorescence

2 INTRODUCTION

2.1 Introduction

The Project is located approximately 40 km south west of the city of Meknès in central northern Morocco (Figure 2-1). The project is owned by Atlas Tin SAS, a joint venture company comprising Kasbah Resources Ltd (75%), Toyota Tsusho Corporation (20%) and Nittetsu Mining Co Limited (5%).

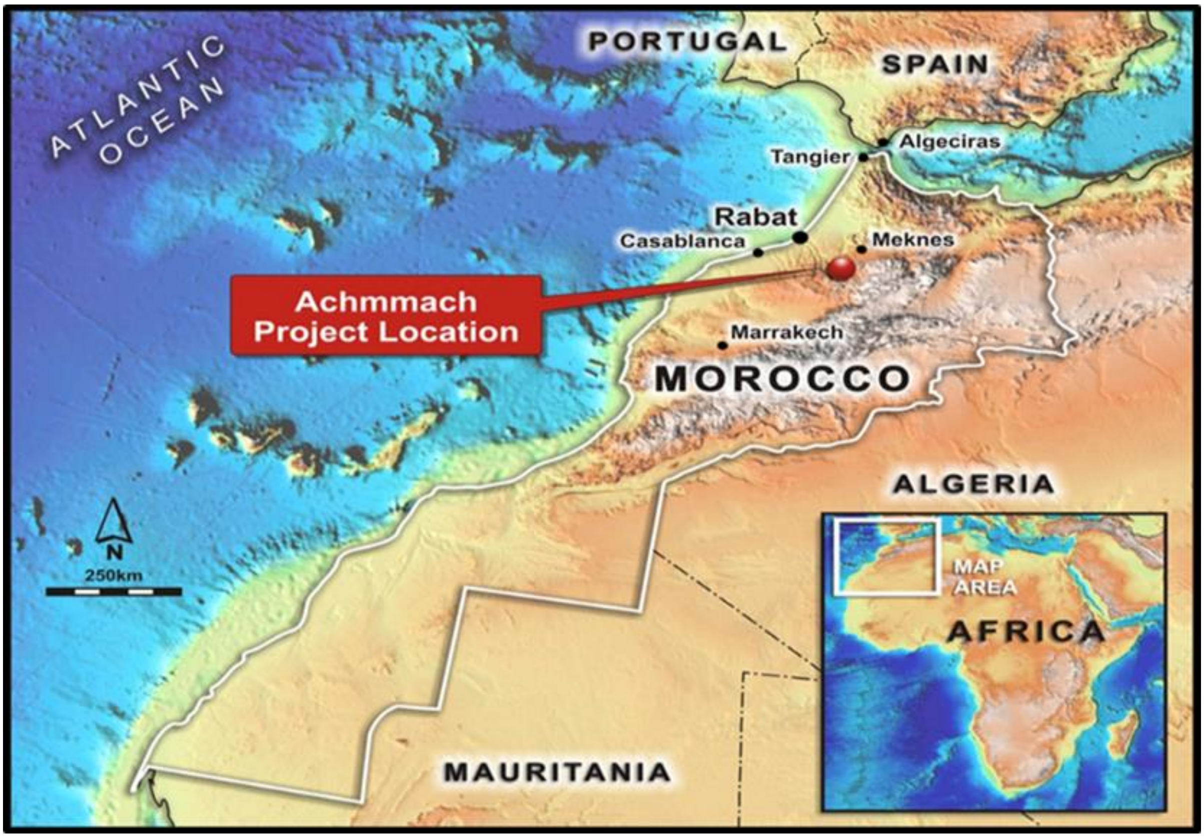


Figure 2-1 Location of the Achmmach Tin Project

This study uses elements of previous studies (2014 DFS (ASX Announcement: 31 March 2014), 2015 EDFS (ASX Announcement: 18 March 2015) and the 2016 SSO (ASX Announcement: 10 August 2016)) and introduces cost and technical enhancements.

2.2 Accessibility, Climate, Local Resources and Physiography

Morocco has well developed national infrastructure including rail, road, sea ports and airports. Access from the capital Rabat to the Achmmach Tin Project is 150 km east along the A2 expressway to Meknès and then 35 km south along a sealed road to Agourai, and a further 20 km south east along an unsealed rural road to the Project site.

The region has a warm and temperate Mediterranean climate. The temperature varies from 5°C/15°C in winter to 18°C/34°C in summer. The winter months are generally much wetter than the summer months and the average annual rainfall is approximately 700 mm.

The Project is located within rugged terrain of the north-eastern part of the central plateau of the Atlas Mountains. The altitude is 1,085 m above mean sea level (amsl) and the nearby Sidi Addi peak has an altitude 1,230 m amsl. The area is characterised by mountain ranges, valleys and plateaus containing pine forests and oak woodland as well as cleared areas used for agriculture.

2.3 History

The Achmmach tin deposit was first discovered by the Moroccan National Office for Mineral Exploration (BRPM) in 1985 by following stream sediment anomalies. In the early 1990's, BRPM conducted several reconnaissance programs such as soil sampling, rock chip sampling, surface mapping as well as a gravity survey. By late 1992, diamond drilling had commenced on targets defined by the early exploration work. Drilling was completed in 2000 with a total of 29 holes over the initial 1.6 km strike length of Achmmach. An 85 m deep exploration shaft and 227 m of development was mined to obtain bulk samples for metallurgical testwork. Three diamond holes were drilled from underground.

In 2002, BRPM produced a resource estimate of 9.57 Mt at 1.09% Sn for a total of 104,000 t of contained metal.

In 2006, Kasbah Resources Ltd entered into an agreement with the Moroccan National Office for Hydrocarbons and Mining (ONHYM) to further test the tin potential of the prospect.

2.4 Study Report

The study report is based on contributions from Kasbah staff supplemented by specialist input from multiple consultants. The contributors to each section are listed in Table 2-1.

Table 2-1 Summary of study contributors

Study Area	Consultants
Geological – Geology and Mineral Resources	QG Consulting Perth
Ore Reserves and Mining Plan	Entech – International Mining Consultants (Perth)
Mining Cost Estimates	International Mining Contractor bids Entech – International Mining Consultants (Perth) Minero – Mining Consultant
Hydrogeology	Golder Associates (UK)
Client Metallurgical Representative	Ore sorting – Tony Parry & Associates Metallurgical – Mike Gunn
Metallurgical Testwork	ALS Global (Perth & Burnie Tasmania) Nagrom (The Mineral Processor) Ore Sorting - Steinert Magnetic & Sensor Sorting High Pressure Grinding - Koeppern Machinery Australia Pty Ltd
Process Plant Design	Lycopodium ADP
Plant Capital	Lycopodium-ADP
Plant Operating Costs	Lycopodium-ADP
Treatment and Refining	Treatment & Refining has been sourced from International Metals traders and Tin Refineries
Infrastructure and Services	Lycopodium-ADP
Environmental Social Impact and Assessment	Artelia Eau & Environment in association with Artelia Maroc
Tailings and Water Storage Facilities	Golder Associates (UK)
Planning and Permitting	Atlas Tin (Morocco)

3 GEOLOGY AND MINERAL RESOURCES

3.1 Introduction

The Achmmach tin deposit is an epigenetic vein-stockwork-breccia style which is associated with a strongly boron enriched paleo hydrothermal system. It is comprised of fine-grained cassiterite with associated minor sulphide minerals in a tourmalinised sandstone/siltstone host. It is interpreted as being hosted by two cross-cutting swarms of tourmaline-altered zones; a series of east-west striking sub-vertical zones described as “feeders” and a stacked series of oblique gently to steeply north-dipping “branches”.

The 1.6 km strike extent of the mineralisation system is hosted by sequences of folded and metamorphosed shales and sandstones. The lodes form a 300 metre wide array across strike with individual lode structures ranging in width from one metre to 30 metres. Tin mineralisation occurs primarily as breccia infill and quartz-cassiterite veins and has been defined in diamond drill holes to a vertical depth of approximately 600 metres below natural surface.

The bulk of the known mineralisation intersected by the current drilling occurs between 1,000 mRL and 700 mRL.

3.2 Conventions

The study is based on metric units; metres and tonnes.

Prior to 2011, the grid system used at Achmmach was universal transverse mercator (UTM), specifically (UTM30N WGS84). However, the mineralisation and therefore the drill hole orientation is oblique to the UTM grid, which resulted in difficulties in interpretation. In 2011, a local grid was introduced with the Easting axis parallel to the mineralisation trend. The local grid has been rotated 20° anticlockwise from UTM, with the RLs unchanged.

Other key conventions and definitions are covered in the Glossary, section 1.21.

3.3 Regional Geology

One of the most dominant features of the Moroccan topography are the Atlas Mountain ranges. These extend from coastal Morocco across the tip of northern Africa through Algeria and Tunisia for approximately 2,000 kilometres and were formed during several orogenic phases (Figure 3-1).

The first tectonic deformation occurred during the Alleghenian orogeny (Paleozoic Era) and resulted from the continental collision of Africa and America producing the Anti-Atlas Mountain belt in south central Morocco. The second phase of deformation (Mesozoic Era) was extension causing rifting and continental separation and producing thick intracontinental sedimentary basins.

The convergent plates of Europe and Africa collided in the final phase of deformation (Cenozoic Era) resulting in uplift and producing the High Atlas Mountains in central Morocco (Ottria et al, 2012). This is the highest mountain belt in Morocco and peaks at over 4,000 metres. Within Morocco, the High Atlas is approximately 800 kilometres long and 50 to 100 kilometres wide.

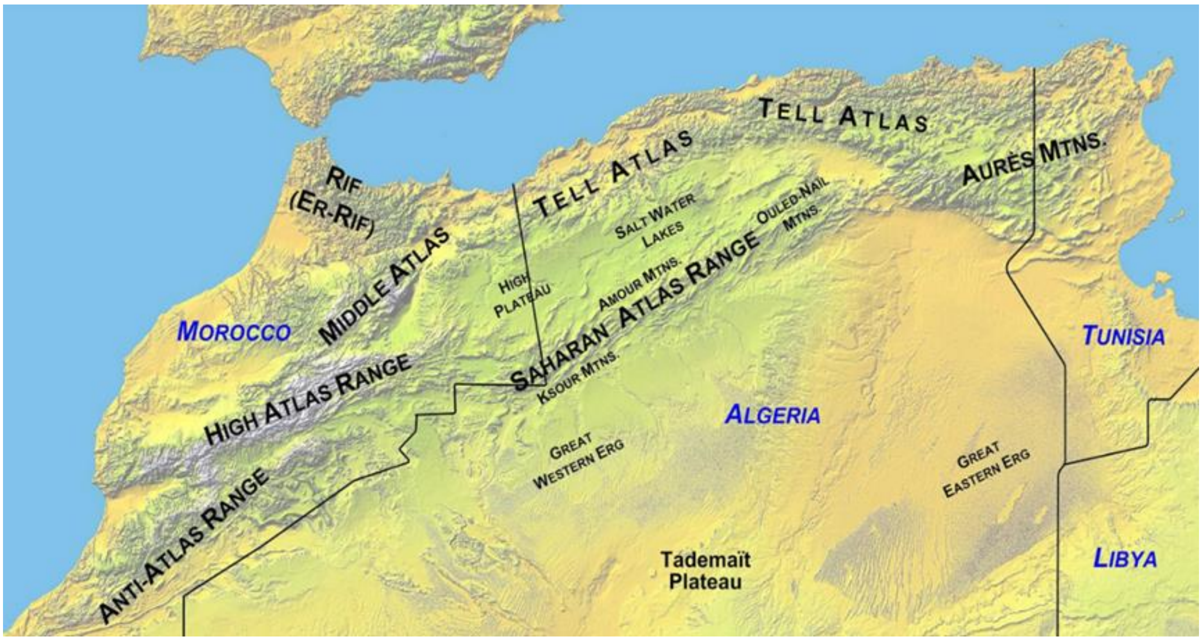


Figure 3-1 Northern Africa Atlas Mountains

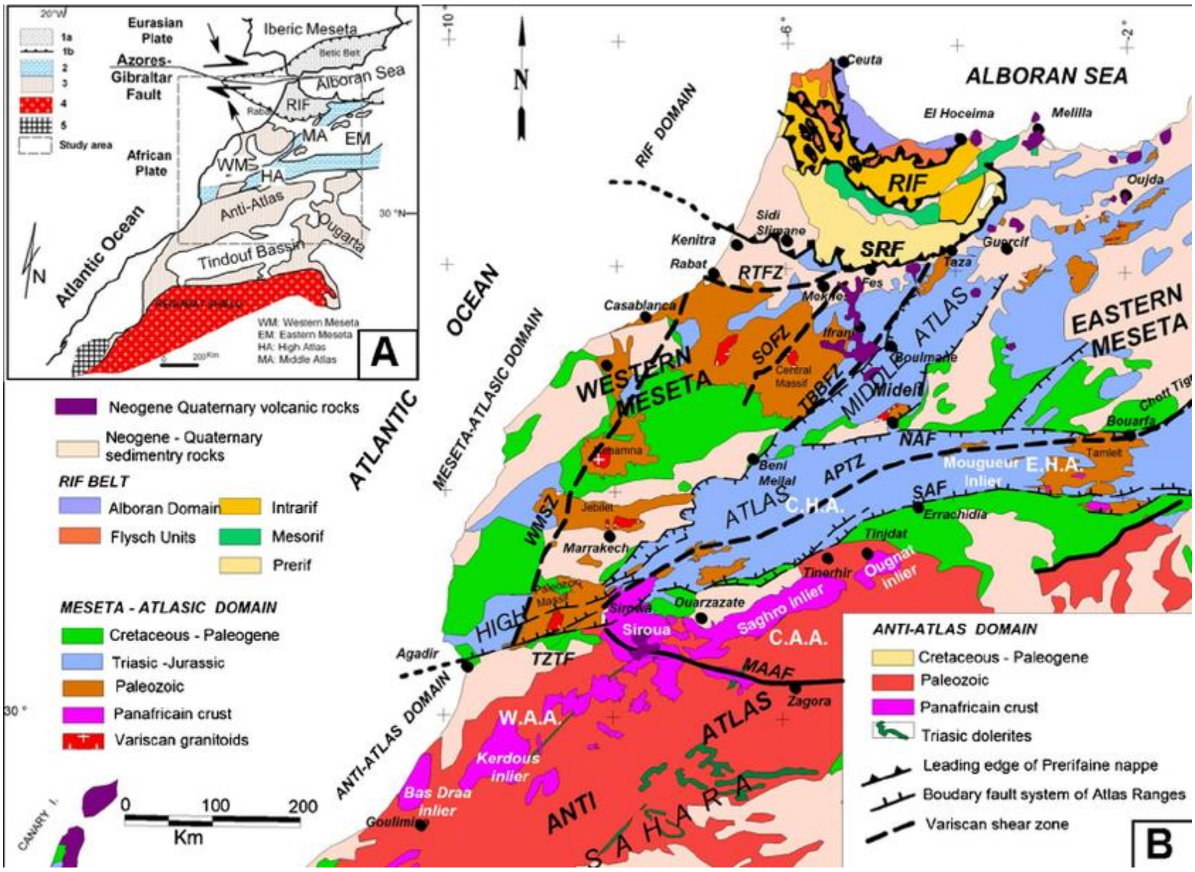


Figure 3-2 Moroccan Geological Domains (after D. Khattach et al, 2013)

Morocco is subdivided into several geological domains, demarcated by regionally extensive palaeozoic faults (Figure 3-2). These domains, from south to north include; the Anti-Atlas and the northern limit of the Saharan domain, the Meseta Atlasic domain and the Rif domain (Khattach et al, 2013). The Palaeozoic basement of the Meseta Atlasic Domain (includes the Moroccan and Oran Meseta, or Western and Eastern Meseta respectively) are exposed as isolated massifs encompassed by Mesozoic – Cenozoic sediments. The Achmmach deposit is in the Central Massif of the Moroccan Meseta and hosted within the turbiditic sediments of the Namurian aged Fourhal Formation.

3.4 Local Geology

The Achmmach tin deposits occur within metamorphosed shales and sandstones of Lower Carboniferous (Mississippian) age. The shales and sandstones occur as a flysch sequence which trend north to north-north-east for at least 20 km. Shale is the dominant rock type. Sandstone units are up to several metres thick but are generally less than 30 centimetres thick with the thicker sandstone units being restricted to the western part of the project area.

The sediments underwent substantial ductile and brittle deformation during the Variscan orogeny after deposition. The resulting bedding is commonly moderate to steeply dipping to WNW. Frequent metre-scale tight parasitic folds are observed over the project area, mostly gently plunging NNE, moderately inclined WNW and ESE-verging.

The regional trend of the strike of the host rocks is north to north-north-east. The Achmmach hill trending 70^{o} east of north translates to a change in strike explained by the occurrence of resistant tourmaline and silica-bearing structures.

The sediments have been intruded by sub-volcanic felsic and mafic igneous rocks. The Palaeozoic sequence has been intruded by Hercynian (Upper Carboniferous-Permian) granite, which outcrops about five kilometres to the west of Achmmach. The host rocks have been overprinted by fine grain black tourmaline over a strike length of at least 1.6 km and a plan width of 200 m to 500 m. This tourmaline alteration corridor is a distinctive feature in the area.

The east-west striking Sidi Addi Fault is the main structural feature at Achmmach and occurs in the northern part of the deposit.

3.5 Deposit Geology

The Fourhal Formation consists of deformed, interbedded pelites and psammites of Lower Carboniferous (Namurian) age. These meta-sandstones and shales comprise a tightly-folded sequence of turbidite beds (Figure 3-3), overprinted by tourmaline alteration within sheared regions and intruded by magmatic sills. Chronologically these sills are interpreted to be pre-mineralisation, as they contain tourmaline alteration and tin mineralisation. The composition of the sills varies and ranges from felsic to mafic. The felsic intrusive (monzogranite) defines the hanging wall of the high-grade mineralisation in the central and western parts of the deposit.

Figure 3-3 Interbedded meta sandstones and shales at Achmmach

Mineralisation is localized in two subparallel ENE striking lodes named the Meknès and Sidi Addi Trends, separated by approximately 500 m (Figure 3-4). The Meknès Trend comprises the largest part of the resource. Mineralisation within the Sidi Addi Trend is referred to as the Western Zone. It is developed within the tourmaline-silica altered metasediments. Tin mineralisation occurs primarily as cassiterite (SnO_2 – S.G 6.8-7.1, 78% Sn) with minor stannite (Cu_2FeSnS_4). It is relatively pure in composition and does not carry significant trace elements (e.g. In, Ga, Ta or Nb).

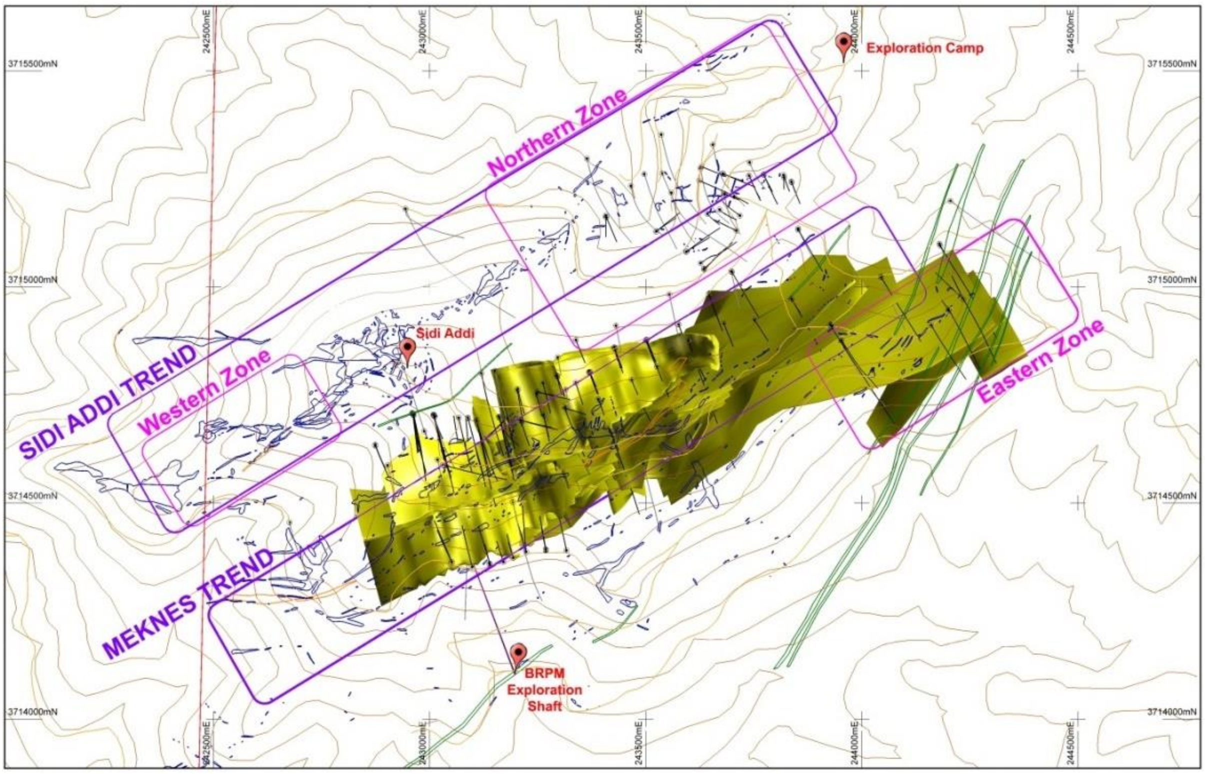


Figure 3-4 Mineralised zones of the Achmmach deposit

Deformation includes repeated folding, faulting and intrusion by suites of igneous sills. Folding produced a moderate to steeply dipping WNW S0 fabric and large parasitic folding. Fold hinges are sheared and thrusted to the ESE along the axial planar cleavages. Milky quartz veins, syntectonic with axial planar cleavages are present through the hinge regions of these folds. There is a pervasive fabric produced by an EW – WSW striking fracture cleavage, thought to have developed during transpressional shearing. This fabric controls the tourmaline alteration and therefore the distribution of tin mineralisation.

Host rocks at Achmmach have undergone extensive hydrothermal alteration, interpreted to be associated with buried granitic intrusions. Two types of structurally controlled alteration occur:

- Quartz-Sericite-Chlorite+/-Pyrite-This is a pervasive, early alteration assemblage that has developed preferentially along faults, cleavages and shear zones.
- Tourmaline-Silica Alteration-This alteration is associated with mineralisation and defines the Meknès and Sidi Addi Trends. These extend for almost 2 km in an ENE direction. The tourmaline alteration, as mentioned, has developed along the EW fracture cleavage but also occupies shear zones, fold hinges, bedding planes and produces fluid-assisted (hydraulic) breccias.

Tin mineralisation at Achmmach occurs as cassiterite in quartz-cassiterite veinlets and stringers, and as disseminations. The veinlets and stringers are commonly narrow, up to a few millimetres thick. Thicknesses up to a few centimetres occur locally. The host rocks were altered by black tourmaline and white quartz before the introduction of cassiterite. This presents as disseminations in black tourmaline and as infill to some breccias.

Taylor (2009) suggested a paragenesis, the stages of which most relevant to the tin mineralisation are:

- tourmaline II and Quartz II occur together and rock containing moderate to strong Tourmaline II mineralisation is host to the tin mineralisation
- quartz-cassiterite ± arsenopyrite, pyrite and chlorite mineralisation contain the tin.

Atlas Tin geologists have noted a general relationship between tin grades and the mode of occurrence of cassiterite as outlined in Table 3-1 below.

Table 3-1 Cassiterite occurrence at Achmmach

Grade Range	Modes of Occurrence of Cassiterite
Up to about 0.5% Sn	quartz-cassiterite +/- sulphide stringers and veinlets not necessarily in tourmaline and quartz-cassiterite stringers and veinlets in tourmaline
About 0.5% to about 2% Sn	as above plus disseminated cassiterite in the interstices of tourmaline
Over about 2% Sn	as above plus wider and higher-grade quartz-cassiterite veinlets and veins or cassiterite as infill matrix of breccia

The mineralised zones occur in discrete envelopes over a strike length exceeding 1.6 km. Close-spaced drilling has identified east-west striking sub-vertical tourmaline envelopes described as feeders and gently to steeply north dipping branches. The tourmaline envelopes have been defined by drilling over a vertical interval of approximately 600 m. The envelopes range from a few metres to over 30 m thick and appear to be relatively continuous in plan, reflecting the continuity of the tourmaline alteration mapped at the surface.

The occurrence and spatial distribution of the cassiterite veinlets in the mineralised envelopes dictates the distribution of tin. The main features of the tin mineralisation shown in Figure 3-5 are described as follows:

- The mineralised envelopes are primarily tourmaline. The envelopes contain the bulk of the tin in the form of quartz-cassiterite veinlets. As with other hydrothermal tin deposits, leakage of tin occurs beyond the envelopes.
- Quartz-cassiterite veinlets are not ubiquitous in tourmaline envelopes. There are parts of the envelopes that are not mineralised with tin. The current geological interpretation does not include any constraint on the location of tin within the envelopes.
- The quartz-cassiterite veinlets are steeply dipping, while the mineralised envelopes are moderate to steeply dipping.

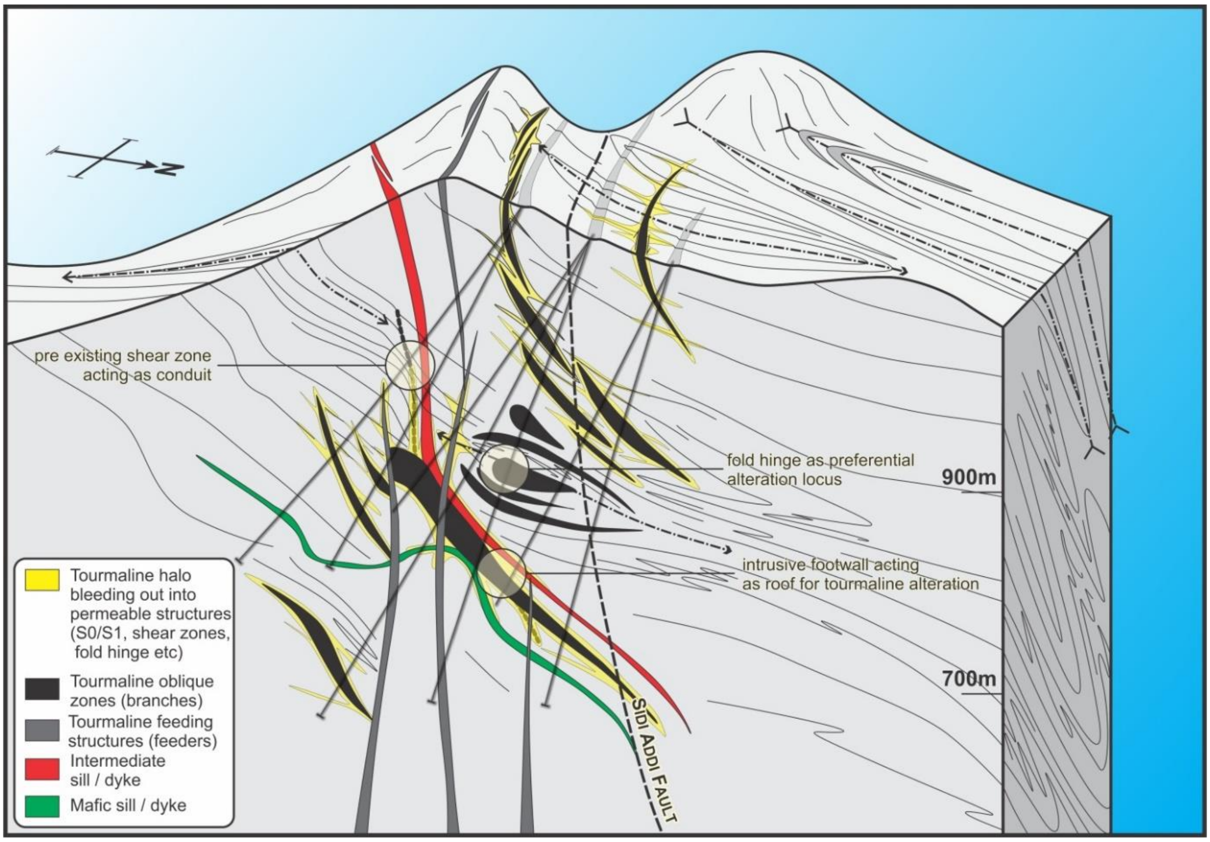


Figure 3-5 View west of the Achmmach geological model sliced at 2450m E (local grid).

Mineralisation occurs as vein, veinlets and disseminations in acicular tourmaline. High grade mineralisation is associated with stockwork arrays, high density sheeted veins and hydraulic breccias.

The mineralisation is developed predominantly within the intense tourmaline-silica altered meta-sediments commonly referred to as lodes. It occurs mostly as veinlets, and centimetre wide veins. Disseminations within the acicular Tourmaline II or the porous sandstone beds are locally important. High grade ore shoots are normally associated with dense fracture / stockwork arrays, bigger and high frequency sheeted veins of up to several centimetres in thickness, and hydraulic breccias as shown in Figure 3-6 and Figure 3-7.

Figure 3-6 Black Tourmaline II –quartz alteration and Milky Quartz I

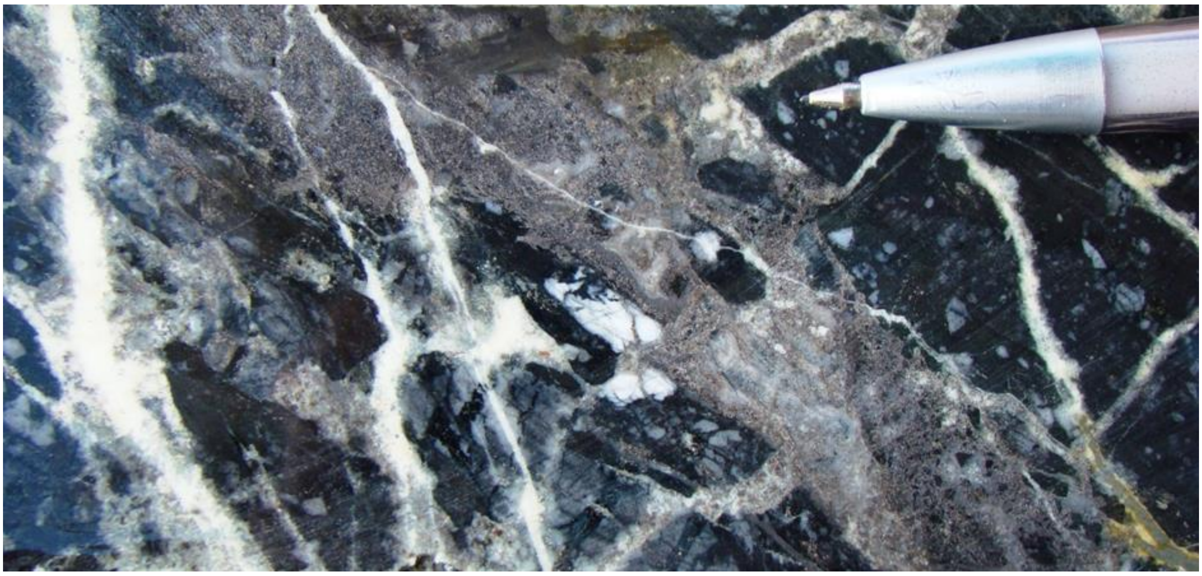


Figure 3-7 Fine grained cassiterite (brownish) as vein infill and dissemination along the edges with late stage cross cutting carbonates (whitish) veins

It appears that the main stages of mineralisation are superimposed on one another (telescoping effect) and as such, no obvious zonation is apparent.

3.6 Data Acquisition

3.6.1 History

The Achmmach tin deposit was first discovered by the Moroccan National Office for Mineral Exploration (BRPM) in 1985 by following stream sediment anomalies. In the early 1990's, BRPM conducted several reconnaissance programs such as soil sampling, rock chip sampling (trenching), surface mapping (1:25000 scale) as well as a gravity survey. By late 1992, diamond drilling had commenced on targets defined by the early exploration work. Drilling was completed in 2000 with a total of 29 holes for 13,405 m drilled over the initial 1.6 km strike length of Achmmach. An 85 m deep exploration shaft (to the 890 mRL) and 227 m of strike development was mined to obtain bulk samples for metallurgical testwork. Three diamond holes were drilled from underground totalling 853 m.

In 2002, BRPM produced a resource estimate of 9.57 Mt at 1.09% Sn for a total of 104,000 t of contained metal (0.5%SN cut-off). This was based on 150 m drill spacing. Table 3-2 outlines the work produced by BRPM.

Table 3-2 BRPM Achmmach exploration summary

Drillhole series	Company explanation	Type of drilling	Start date	End date	No of holes	Max Depth (m)	Total (m)
S01-02, S04-11	BRPM – Northern Zone test drilling, 50 m space grid	DD	12/1991	10/1993	10	635	3,907
S03, S23	BRPM – structural holes, deep target	DD	04/1992	06/1996	2	1,083	1,807
S12-22, S24-29	BRPM – Meknès Zone test drilling, wide space 150 m grid	DD	03/1993	07/1997	17	588	7,690
SF1-3	BRPM – underground holes	DD	06/1997	10/1997	3	329	852
Total					**32**		**14,257**

In 2006, Kasbah Resources Ltd entered into an agreement with the Moroccan National Office for Hydrocarbons and Mining (ONHYM) to further test the tin potential of the prospect. The works included:

- surface mapping to 1:1000
- soil sampling at 160 m x 80 m grid (1,218 samples)
- rock chip sampling (447 samples)
- ground magnetic surveys in 2008 and 2009
- diamond drilling in 2007 comprising 292 holes drilled for approximately 105,000 m (Table 3-3)
- paragenetic studies in 2008, 2009 and 2012
- structural studies on oriented core in 2009, 2010, 2011 and desktop work in 2012
- PhD research on mineralisation textures, fluid inclusions and model of emplacement.

Table 3-3 Kasbah exploration history (DFS 2014)

Company description	Type of drilling	Start date	End date	No of holes	Max depth (m)	Total (m)
Early Eastern Zone test drilling	DD	Nov-07	Apr-08	17	465	4,192
Meknès Resource Drilling	DD	Apr-08	Mar-13	66	591	25,784
Fez Zone, lateral lode continuity and NS high strain zone testing	DD	Jan-11	Jun-12	10	524	3,979
"Gap" Zone resource drilling, 80 and 40 m infill	DD	Mar-11	Jul-12	59	570	24,783
Meknès Eastern Zone resource drilling, 40 m spaced sections	DD	May-12	Apr-13	61	629	25,579
"Gap" Zone resource drilling, 20 m infill	DD	Oct-12	May-13	42	551	16,499
"Eastern Zone Shallow" resource drilling	DD	Feb-13	May-13	15	205	2,088
"Western Zone Shallow" resource drilling	DD	Jun-13	Nov-13	22	136	2,005
Total				**292**	**3,671**	**104,910**

3.6.2 Drilling

Meknès Trend

Approximately 325 diamond holes have been drilled on the Meknès Trend. Of these, 271 holes have been used to estimate the mineral resource. This totals to 109,618 m and includes:

- 17 BRPM surface diamond holes for a total of 7,690 m
- 3 BRPM underground diamond holes for a total of 853 m
- 251 Atlas Tin surface diamond holes for a total of 101,075 m.

Western Zone (Sidi Addi Trend)

Significantly less work has been conducted on the Sidi Addi Trend (Western Zone) than the Meknès Trend. From June 2013 to September 2014, 35 shallow holes were drilled into this area. These were used to estimate this portion of the resource. The holes produced HQ sized core and total metres were 4,550 m. Four holes from this data set were drilled using triple tube HQ3 and were also used for geotechnical assessment.

3.6.3 Meknès Trend

Analysis of Data Prior to 2008

Datasets prior to mid-2008 i.e. BRPM holes S01-S29 (1991 – 1996), SF1 – SF3 (1997) and Atlas Tin holes prefix AD001-AD025 were reviewed in earlier resource estimates. This was done because the uncertified standards from the Atlas Tin dataset used in the Reminex laboratory in Morocco showed a conditional bias (high). However, check assays (on pulped samples retrieved from Reminex) were sent to ALS, in Seville, Spain and showed very good correlation between the two laboratories. The check assays on the BRPM data showed a slightly low but inconclusive bias and it is considered that the volume of the newer Atlas Tin drilling will have negated any possible and slight bias in the small historic dataset.

Analysis of Data from 2009 to mid-2010

Quality control practices for samples submitted from 2009 to mid-2010 included the use of three uncertified standards. Although these weren't certified they were assayed nine times each at the Wheal Jane Laboratory in Cornwall in England to yield an 'accepted' value. The value was verified by laboratories in both Australia and Morocco. The data was independently reviewed confirmation that the results were not biased and are reliable for use in the mineral resource estimate. The assay laboratory used during this period was ALS in Seville, Spain, with some additional work at ALS in Perth, Australia.

Analysis of Data from mid-2010 – mid 2012

Additional QAQC practices introduced during this period include the submission of blanks, lab repeats and check samples as well as three Certified Reference Materials (CRM's). The three CRM's are shown in Table 3-4 below and are sourced from Ore Research and Exploration (ORE) in Melbourne, Australia. These are representative of varying orebody grades i.e. low, medium and high grade. The standards are certified for an additional nine elements. Previous authors have reviewed the data from this period and confirmed that it was satisfactory and acceptable for use in the mineral resource estimate. The assay laboratory used during this period was ALS in Seville, Spain, or ALS in Kirkenes, Norway with some additional work done at ALS in Perth, Australia.

Table 3-4 Certified values and confidence limits for Sn standards

Name	Certified Value	Standard Deviation	1SD Low	1SD High	2SD Low	2SD High	3SD Low	3SD High
OREAS 140	0.1777	0.0042	0.1735	0.1819	0.1693	0.1861	0.1651	0.1903
OREAS 141	0.6312	0.0259	0.6053	0.6571	0.5794	0.683	0.5535	0.7089
OREAS 142	1.08	0.04	1.04	1.12	1.00	1.16	0.96	1.2

Analysis of Data from 2012

In June 2012, the assaying was switched from the ALS laboratories in Spain and Norway to the ALS laboratory in Loughrea, Ireland.

Standards

To test for accuracy during this period, a standard sample was introduced after 20 regular samples. Control charts for the three standards are located below in Figure 3-8 to Figure 3-10. The control chart for OREAS 140 (low grade standard) shows no bias and data mainly falls within expected ranges. However, the control charts for the other two standards highlights a small conditional bias with both of the higher-grade assays returning lower than expected results. The majority of assays from OREAS141 returned values 0.024% Sn lower than the certified value whereas OREAS142, the highest grade value consistently returned values 0.04% Sn lower than the certified value.

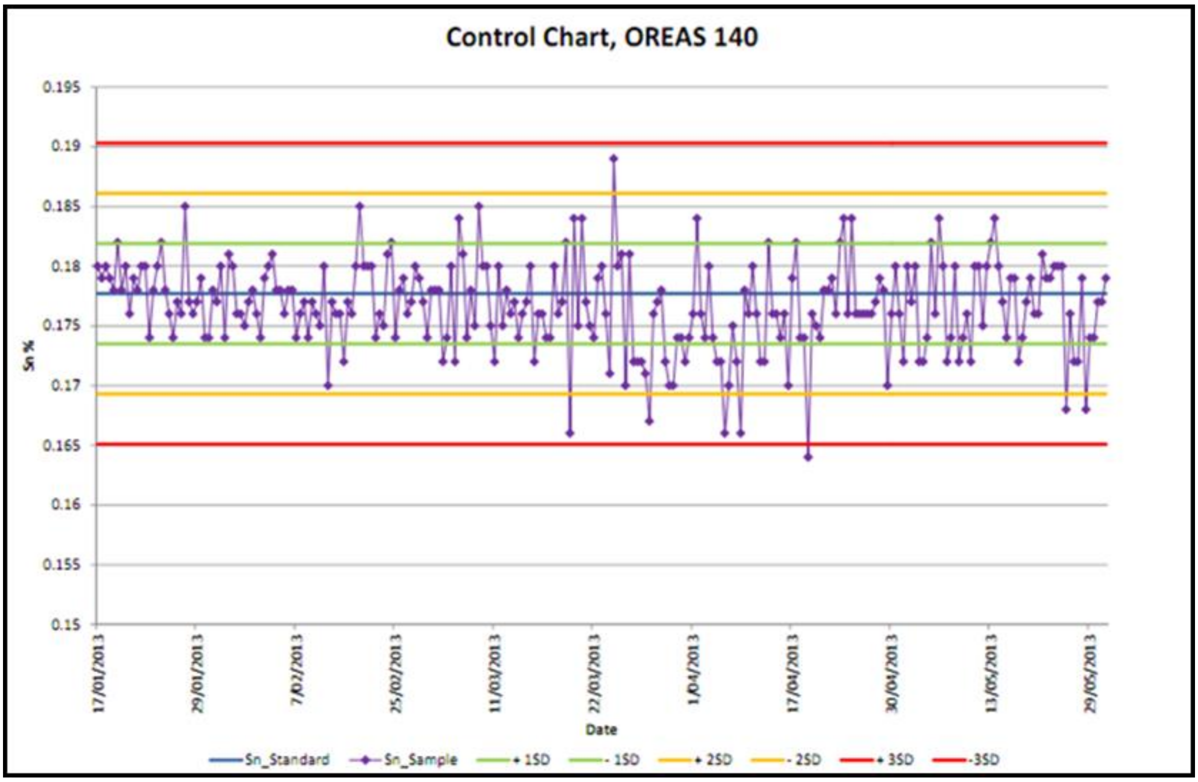


Figure 3-8 Results for Standard OREAS 140

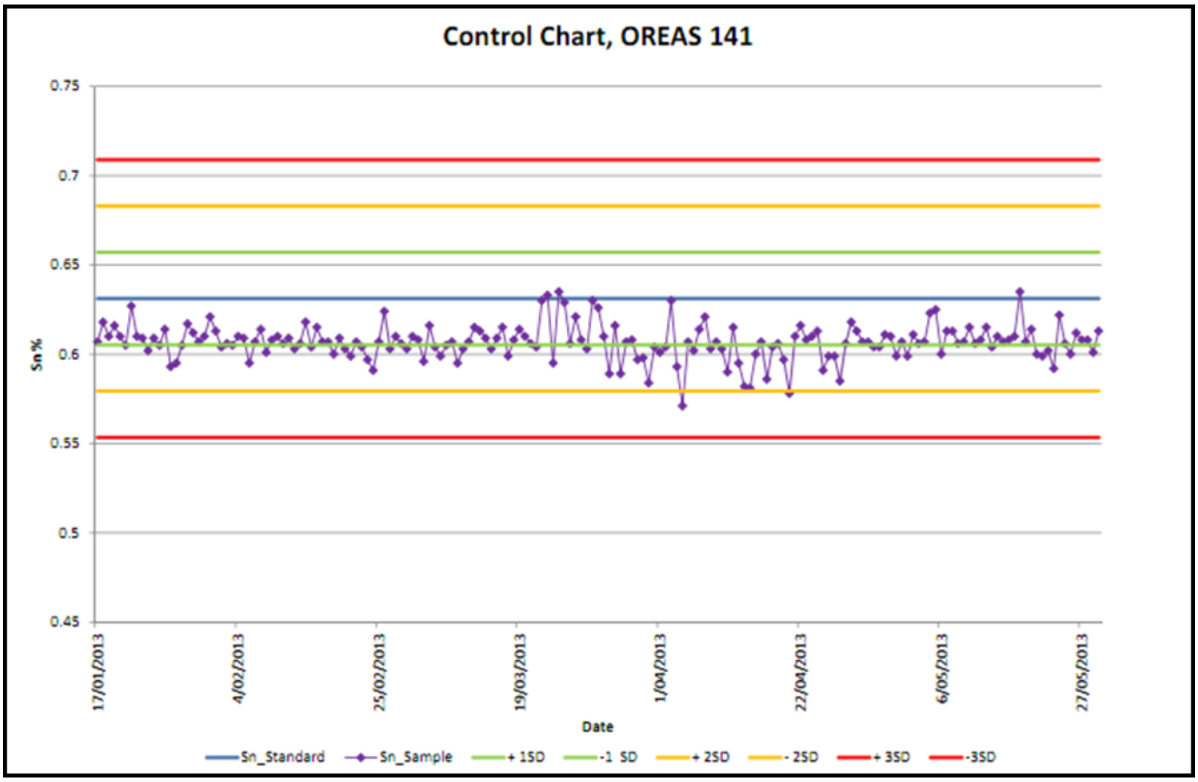


Figure 3-9 Results for standard OREAS 141

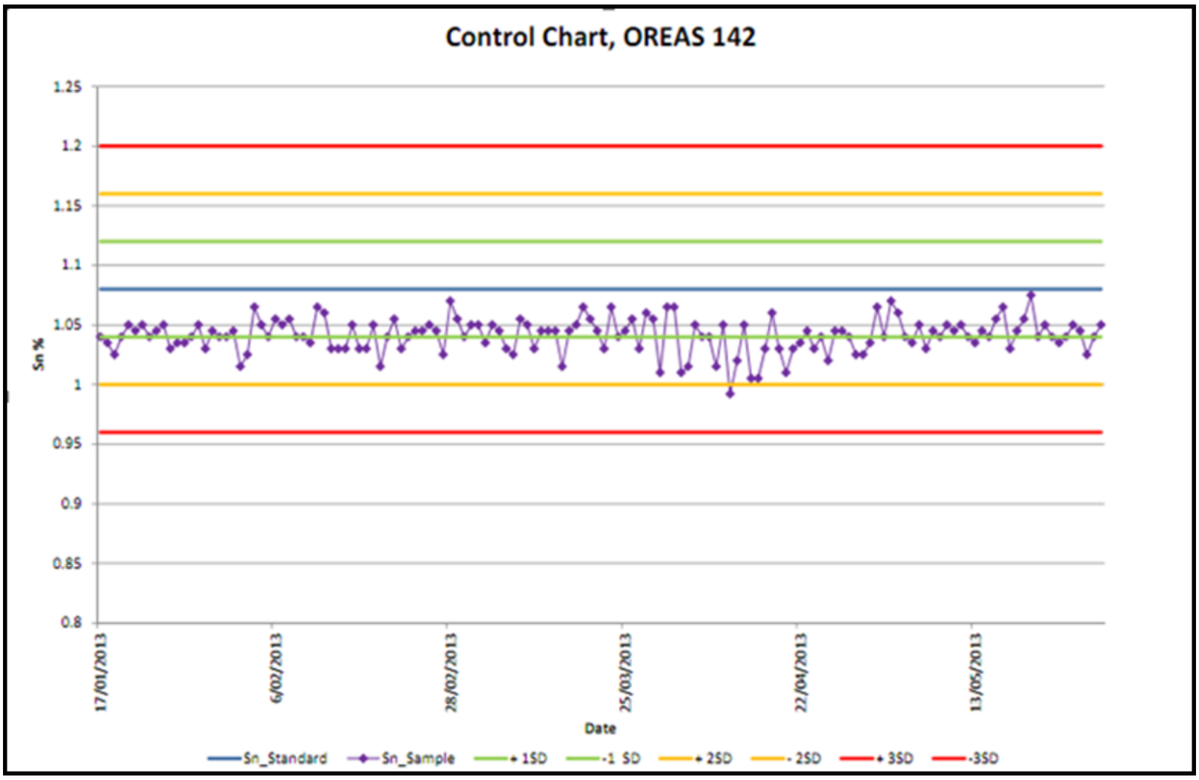


Figure 3-10 Results for standard OREAS 142

Blanks

From May 2011 onwards, blanks were routinely inserted into all exploration programs at a rate of one blank per 50 samples. The blanks are locally sourced barren quartzite.

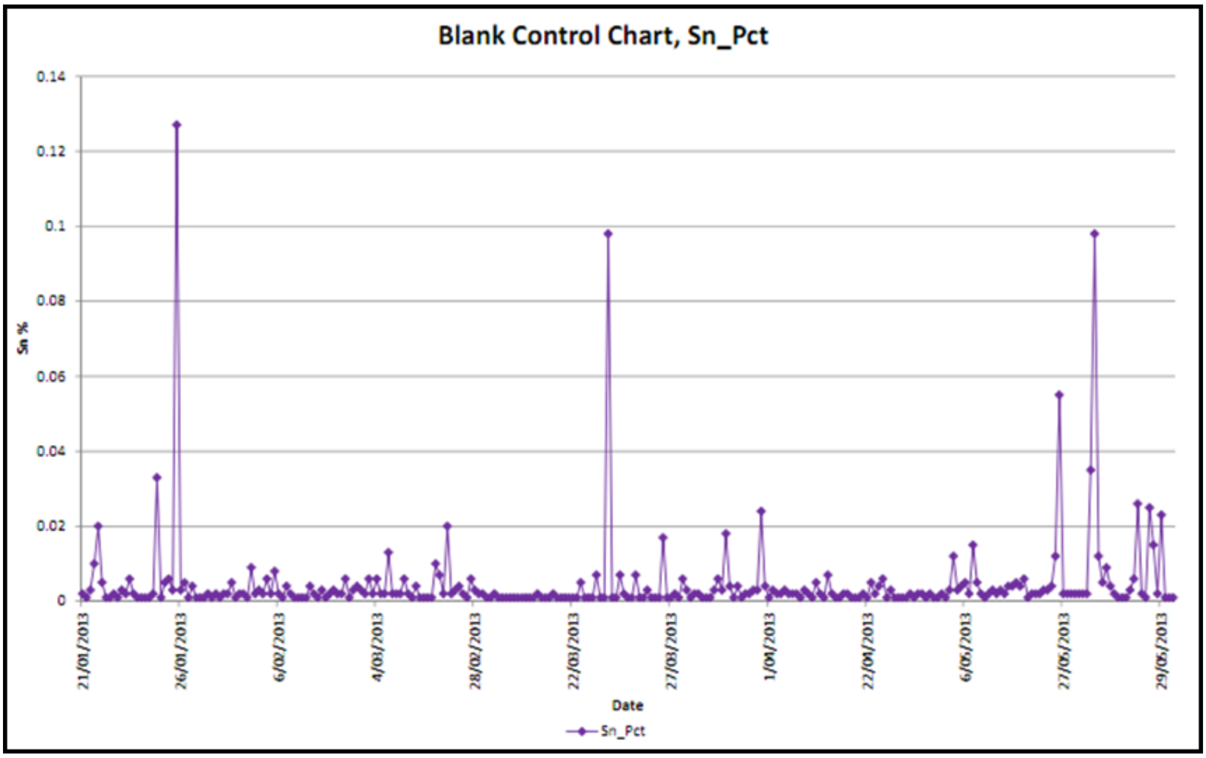


Figure 3-11 Results for blanks control chart

For this data period the blanks demonstrated an acceptable level of contamination with isolated and low grade events (Figure 3-11).

Duplicates

To test the precision of the assay data, two types of sample duplicates were conducted routinely from the beginning of 2010.

The first duplicate is a coarse crush duplicate, collected during the on-site crushing stage. This test determines the precision of the site sample preparation stage of the assaying process. The second type of duplicate was a pulp repeat, assayed at the same time as the original sample. This checks the analytical precision. This is displayed in the following scatterplots and precision charts for both types of pulp repeats and coarse crush duplicates (Figure 3-12 to Figure 3-15).

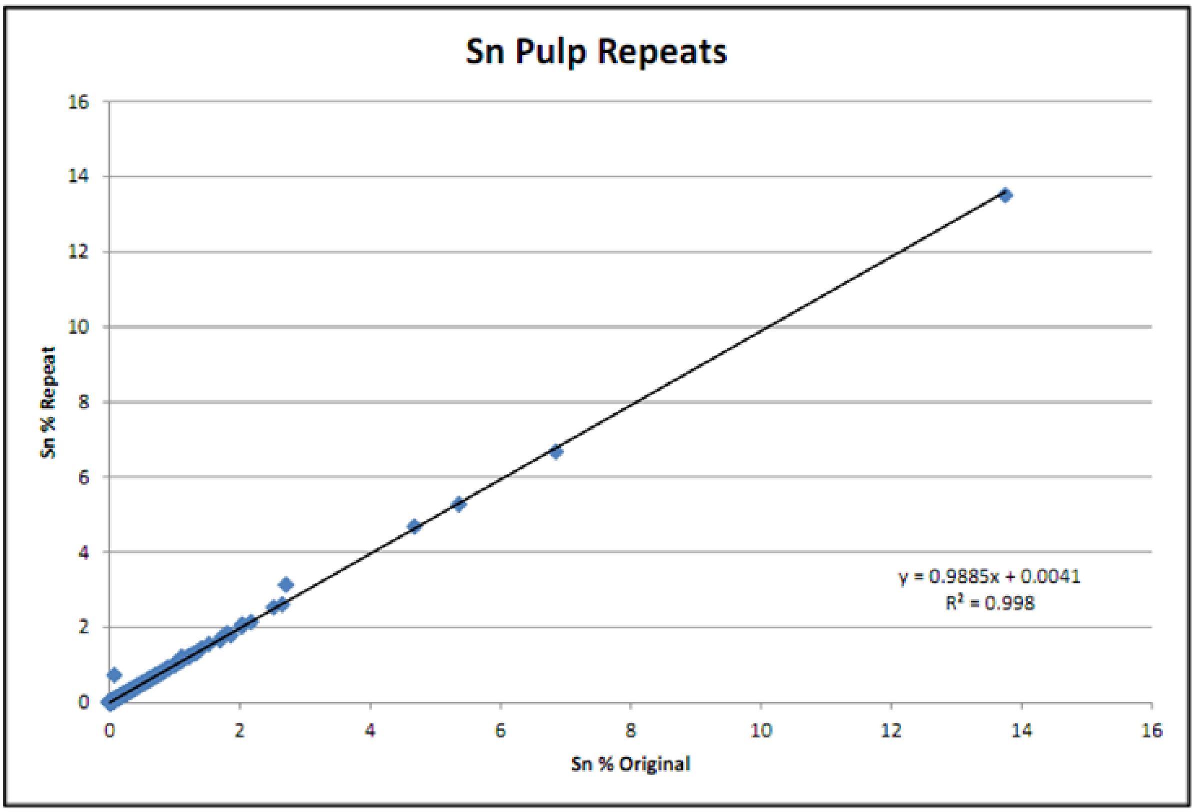


Figure 3-12 Scatterplot Sn original vs. pulp repeat

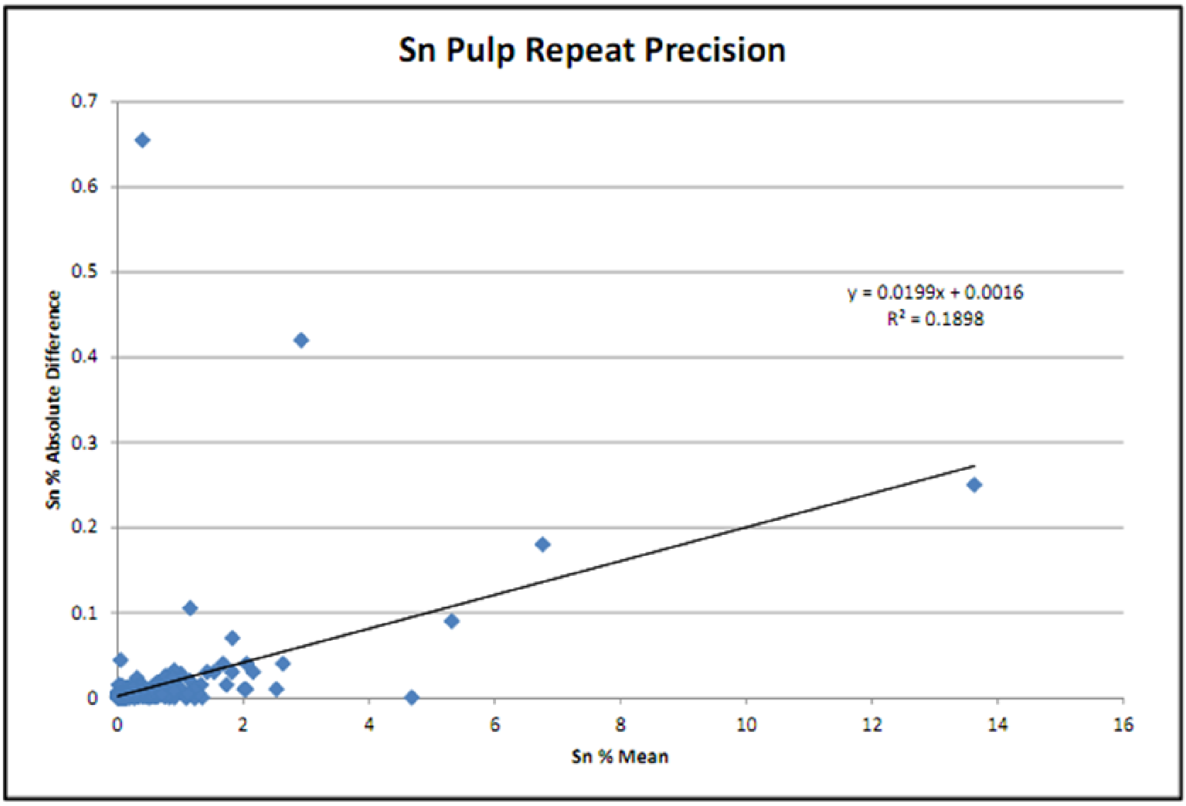


Figure 3-13 Sn repeat precision chart

The charts show that for the pulp repeats the correlation between the original and repeat assay are strong, indicating that the analytical precision is sufficient.

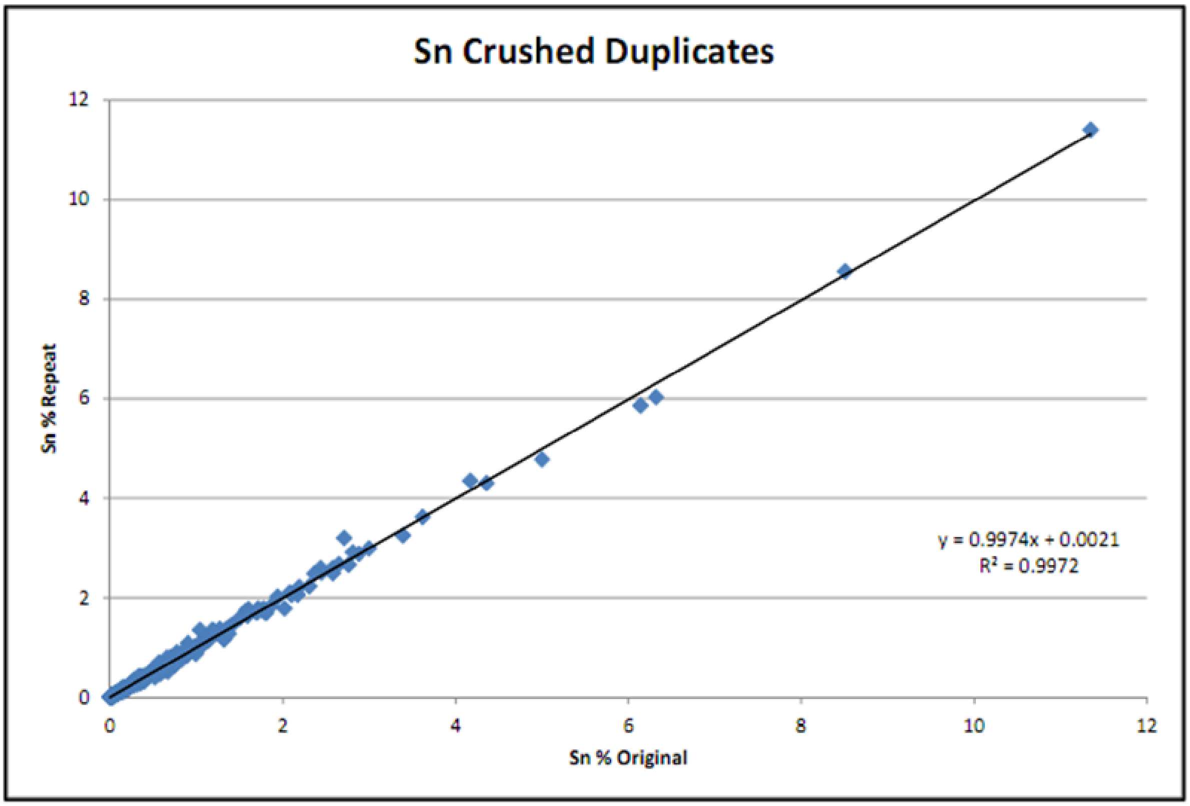


Figure 3-14 Scatterplot Sn original vs. coarse crush duplicate

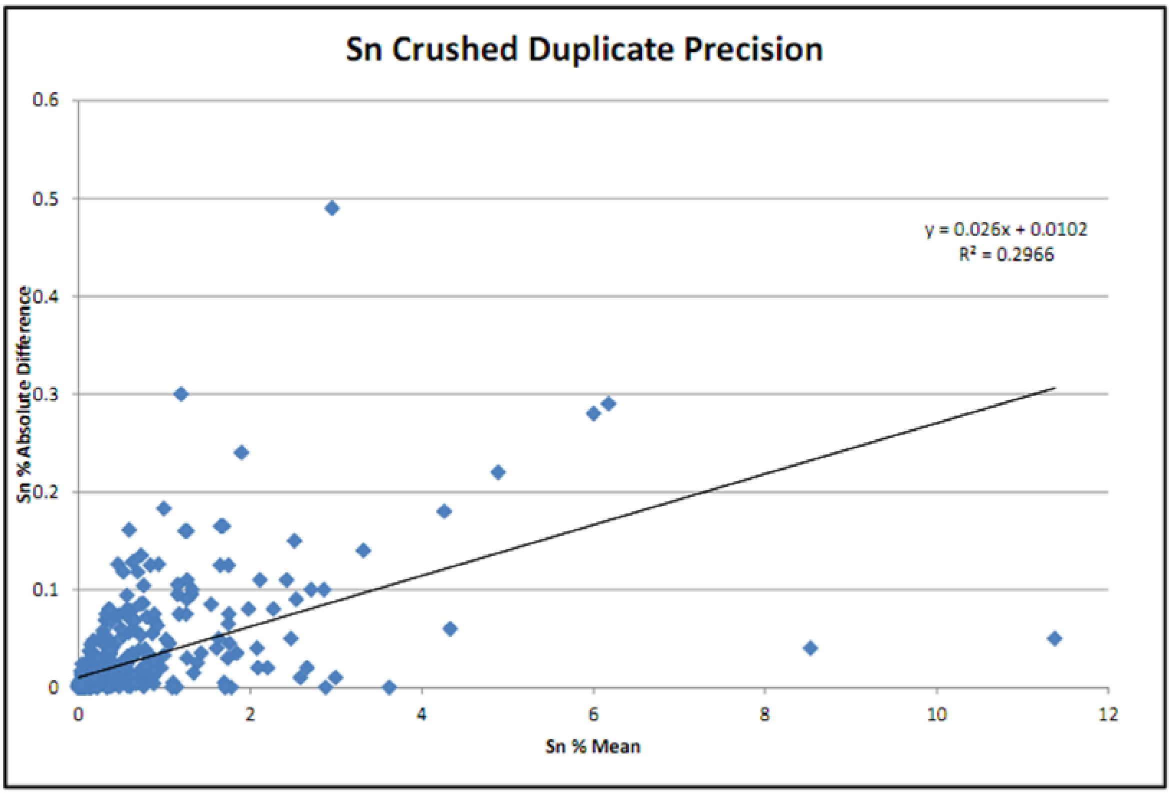


Figure 3-15 Sn original vs. coarse crush duplicate precision chart

The charts for the coarse crush duplicates are not as precise as the pulp duplicates. This is because the point in the sampling chain has a significant effect on the precision that can be expected. By reducing the particle size of the material, the variability between the repeat samples is reduced. That said, the magnitude of the error is small and the data is acceptable and importantly, shows no bias.

3.6.4 Umpire Laboratory Checks

To test the laboratories precision, several sample pulps were sent to an independent umpire laboratory (Amdel-Bureau Veritas) in Perth, Western Australia. The charts show a high level of correlation indicating the ALS laboratory in Ireland was producing acceptable results (Figure 3-16 and Figure 3-17).

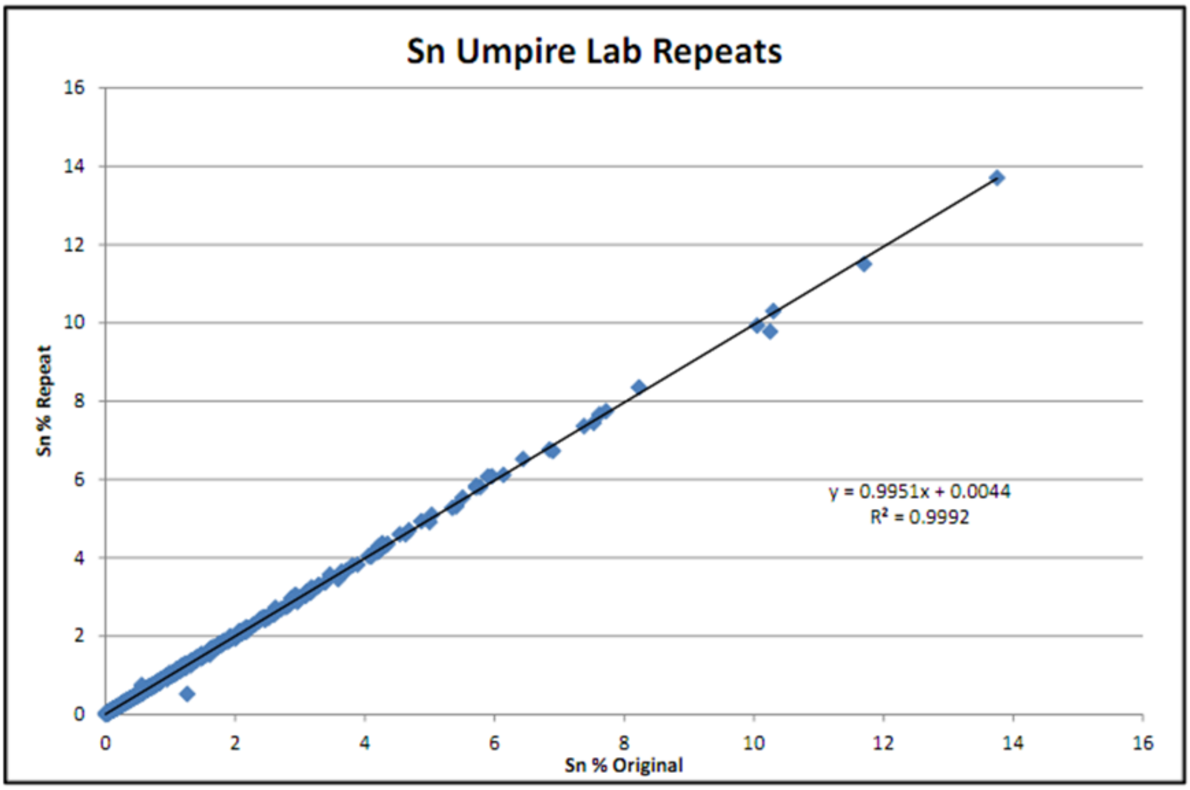


Figure 3-16 Umpire laboratory repeat scatterplot

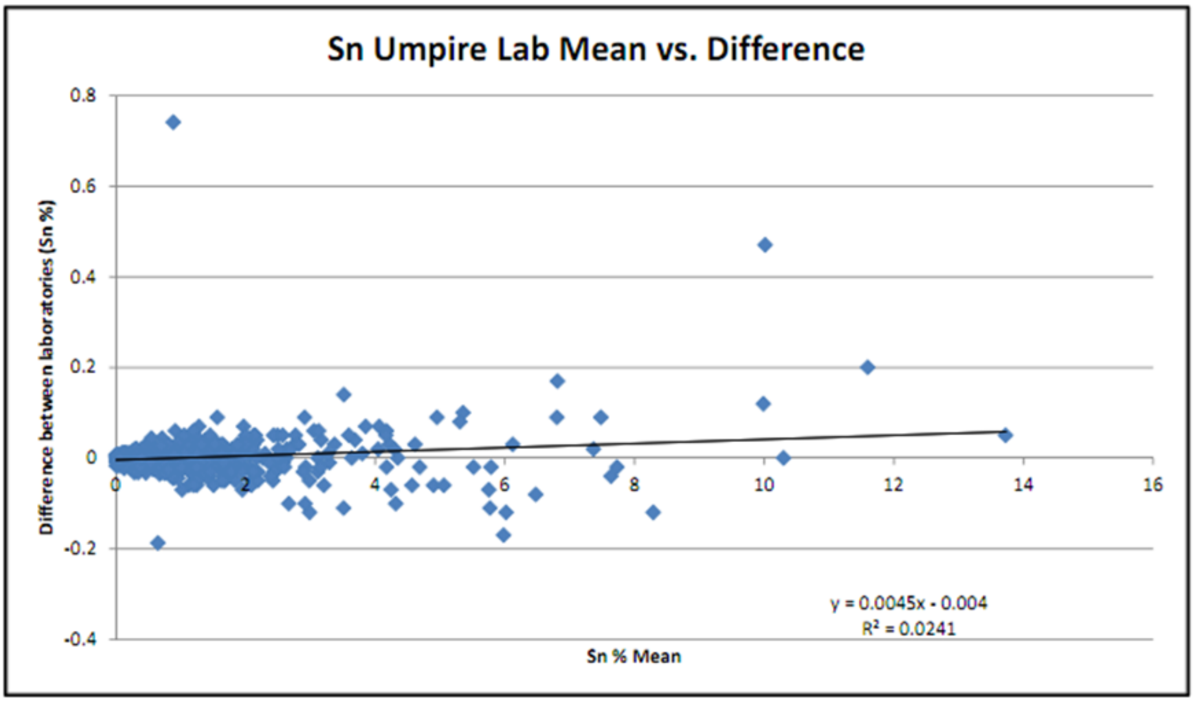


Figure 3-17 Umpire lab precision pairs plot (difference vs. mean)

3.6.5 Western Zone (Sidi Addi Trend)

Standards

The same three CRM's used for the Meknès Trend were also used to the Western Zone. Standards were also inserted at a rate of 1 in 20 samples. Control charts for each of the CRM's are presented below (Figure 11.11 to Figure 11.13).

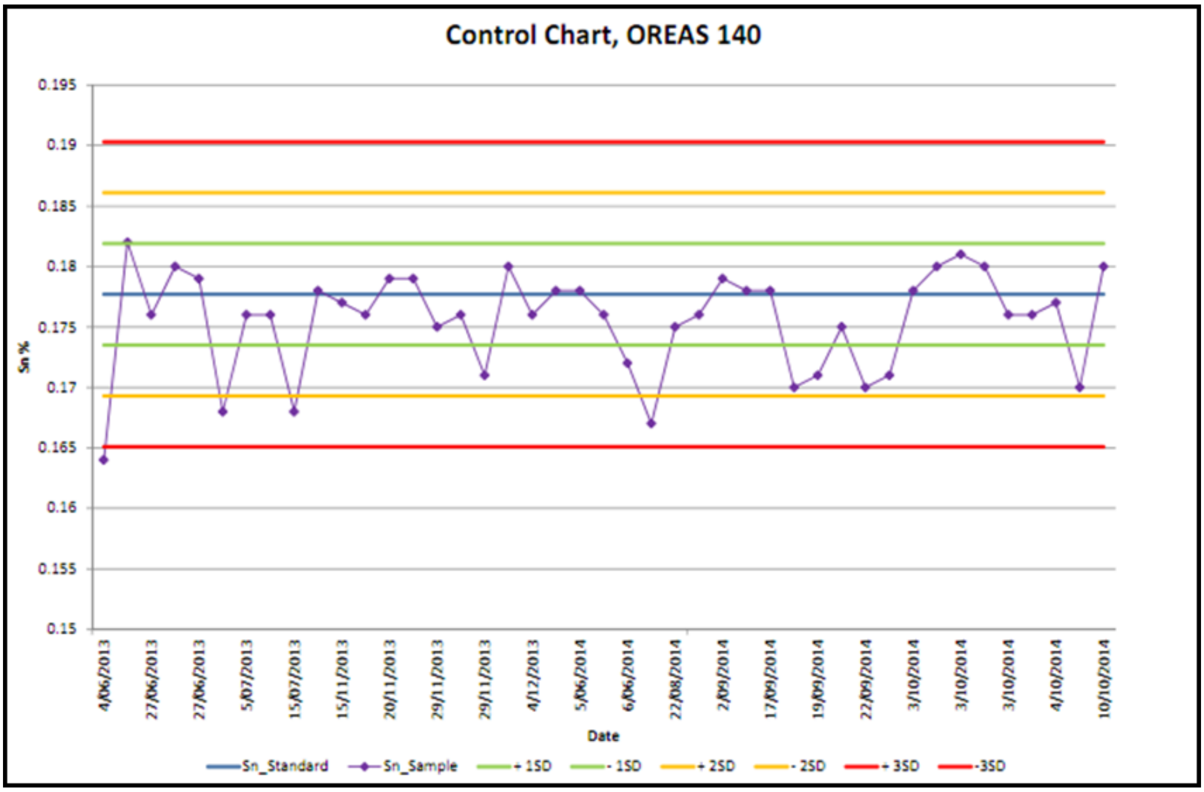


Figure 3-18 Results for Standard OREAS 140

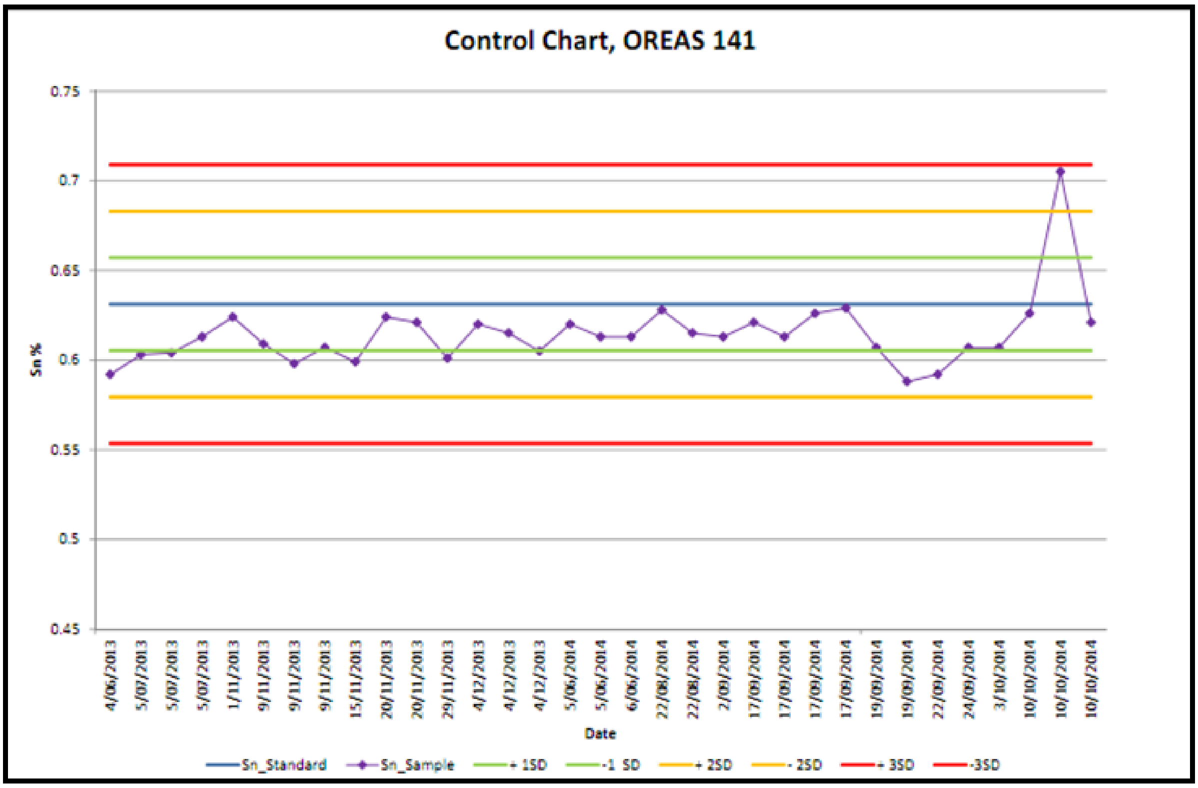


Figure 3-19 Results for Standard OREAS 141

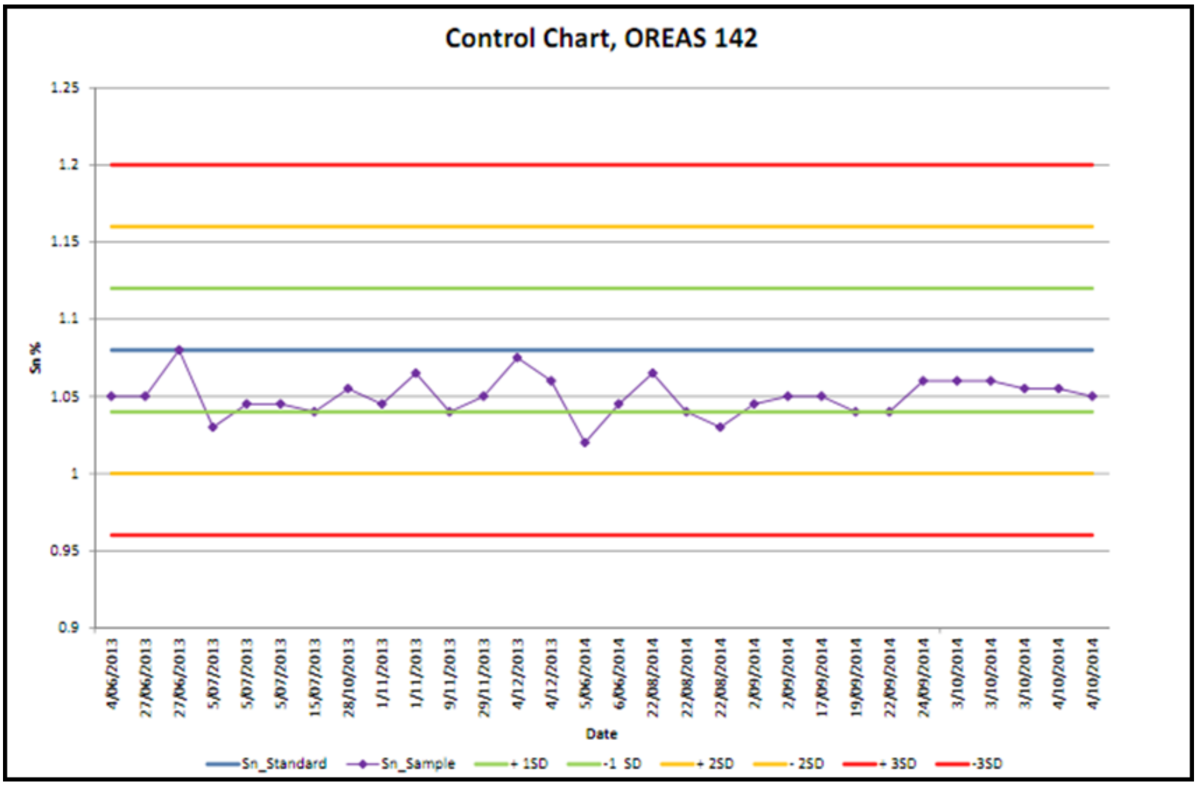


Figure 3-20 Results for Standard OREAS 142

All three CRMs show a very slight low bias, which is more evident in the two higher grade standards (Figure 11.11 to Figure 11.13). This may lead to a slightly conservative estimate at these grades.

Blanks

Blanks for the Western Zone drilling program were inserted at a rate of 1 in 30 samples. The blank material is the same barren quartzite that was used previously for the Meknès Trend (Figure 11.14).

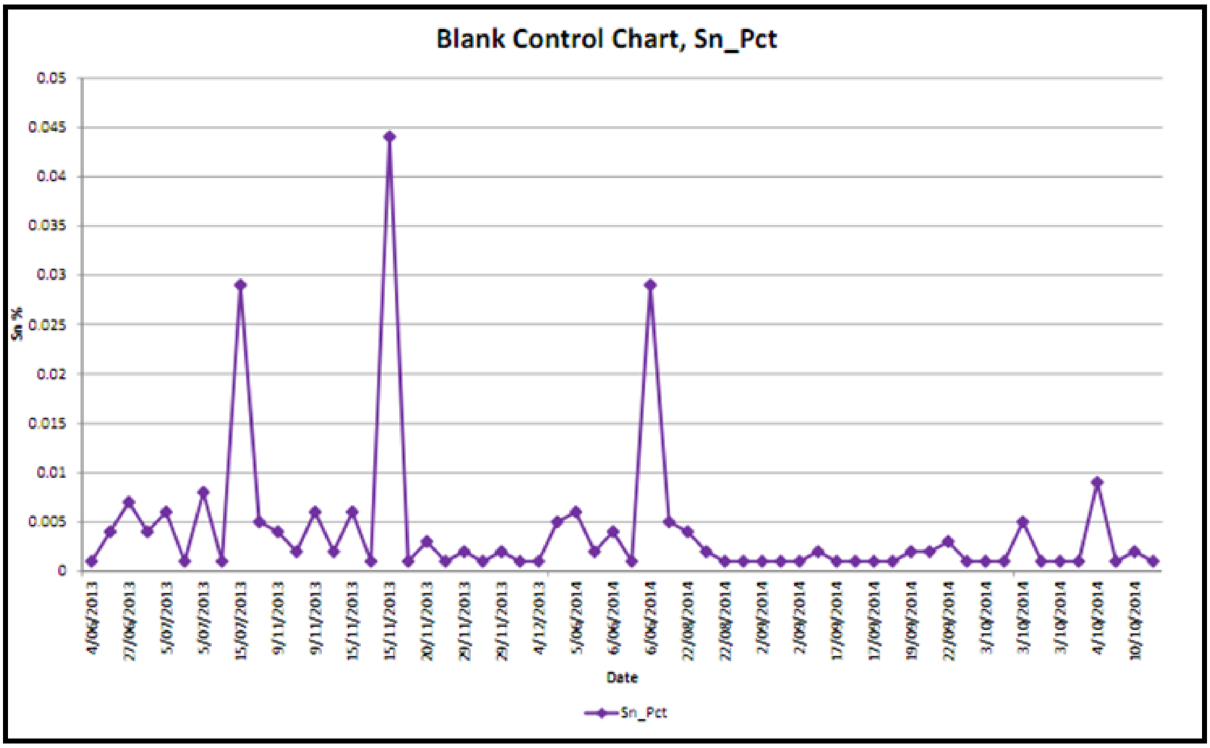


Figure 3-21 Blanks Used in Western Zone

The chart shows some potential very low grade contamination occurs in three holes. The highest spike (on the 15/11/13) occurred following a high grade Sn sample (>8%Sn). These pulps were re-assayed and similar results ensued. However, the magnitude of the error is very small and unlikely to cause significant misclassification or error in resource estimates.

Duplicates

The same two types of sample duplicates used for the Meknès Trend were used for the Western Zone. The first duplicate is a coarse crush duplicate, collected during the on-site crushing stage. This duplicate is used to test the site sample preparation stage of the assay process. The second type of duplicate was repeat assay of pulps which tests the analytical stage of the assaying process. Scatterplots and precision pairs plots (Figure 3-22 to Figure 3-25) of the data show reasonable correlation for both duplicates types with the precision being not as good for the coarse crush duplicates. This is due to the cumulative error during the sampling and assaying process. The error is very small and is not considered to impact the data quality.

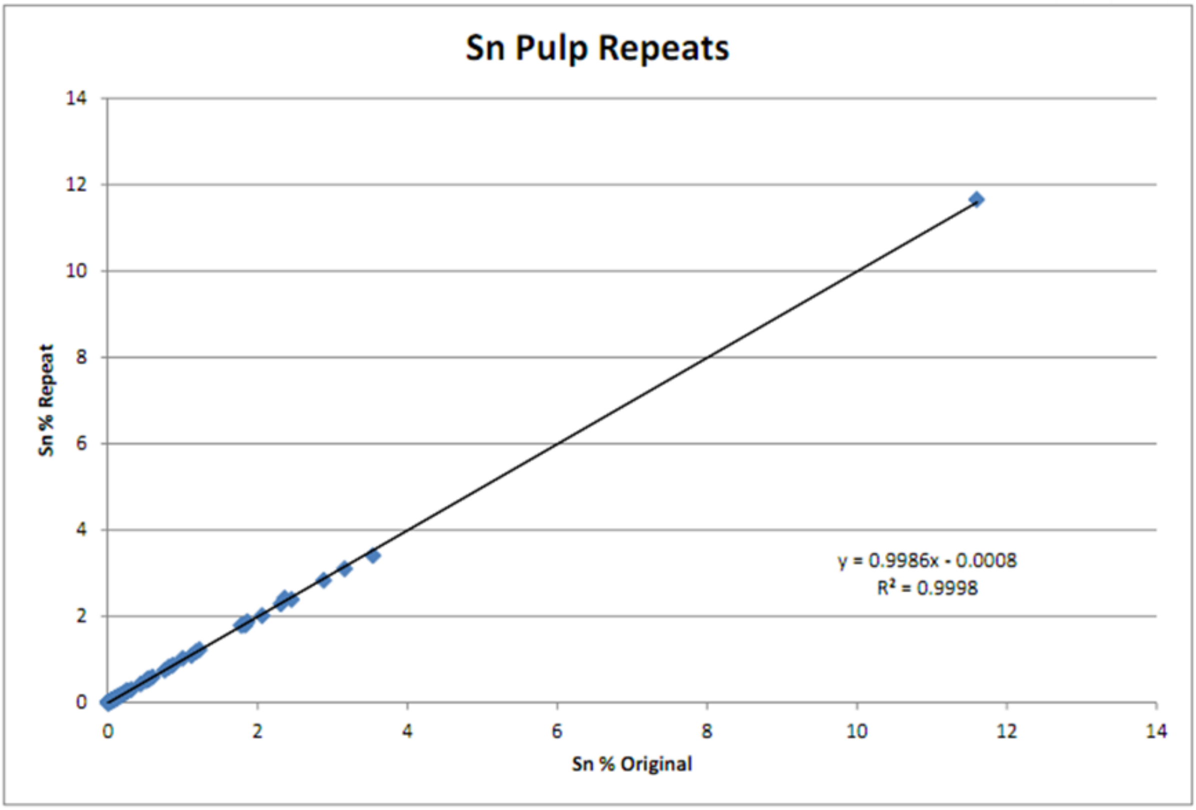


Figure 3-22 Scatterplot Sn original vs. pulp repeat

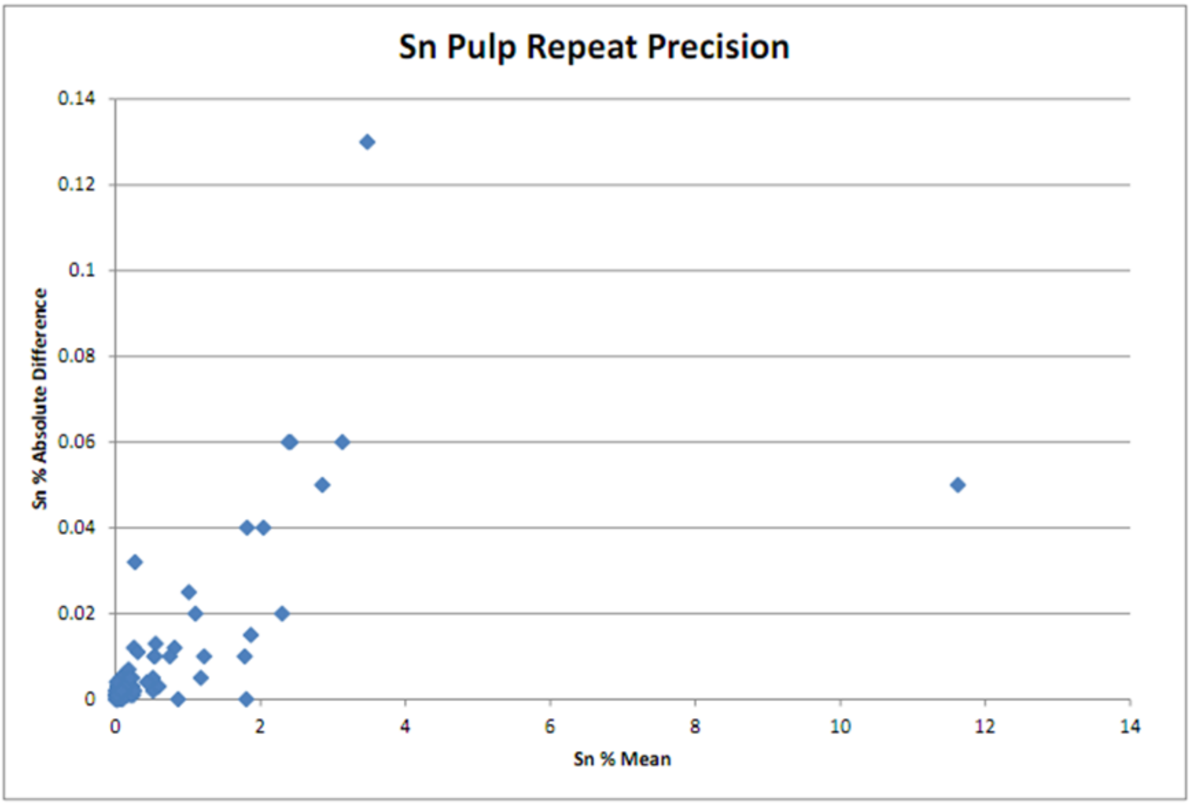


Figure 3-23 Sn repeat precision chart

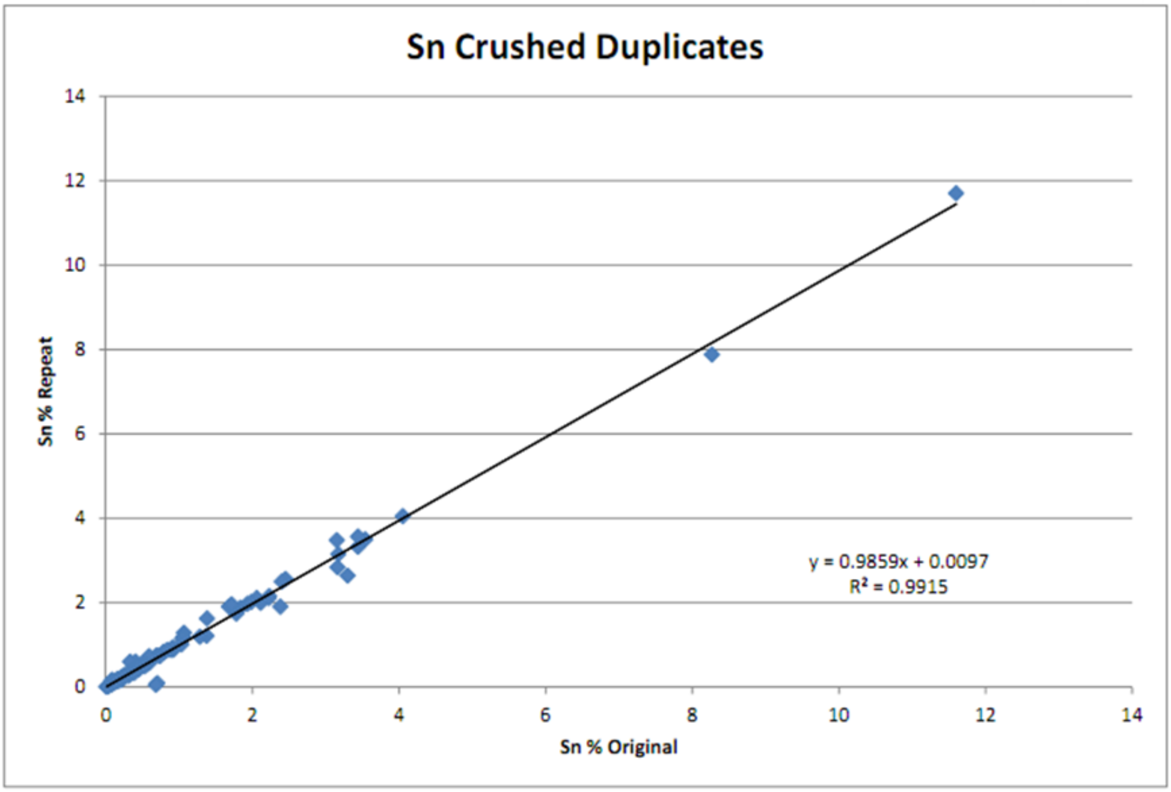


Figure 3-24 Scatterplot Sn Original vs. Coarse Crush Duplicate

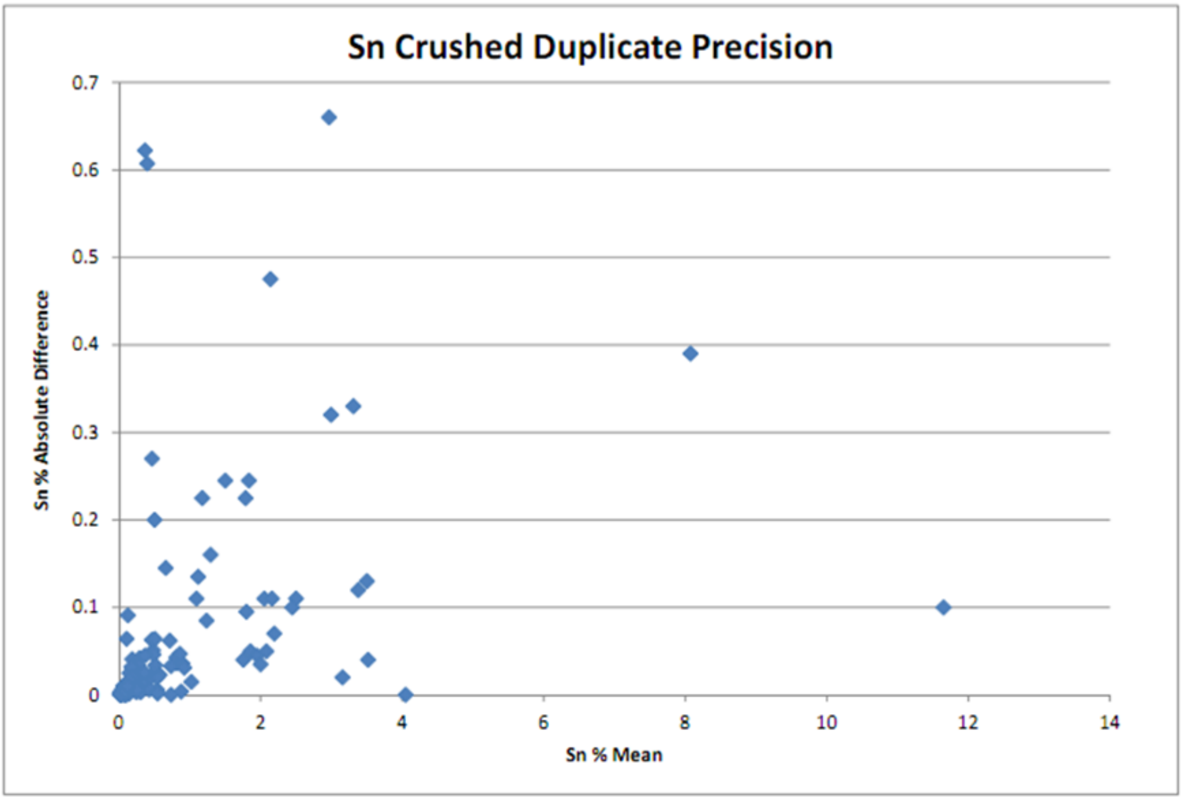


Figure 3-25 Sn Original vs. coarse crush duplicate precision chart

3.6.6 Downhole and Collar Surveys

All drill holes have been downhole surveyed. However, the method and therefore accuracy for the S and SF prefix holes drilled from 1991 – 1997 by BRPM is unknown. The AD prefix holes drilled by Kasbah (after 2007) used either multi shot (2007 – 2008) or single shot (2009-2013) Reflex survey instrument. Measurements were taken every 50m prior to 2010 and then at 25m intervals from 2011 onwards.

Collar positions for the Kasbah holes were initially set out using a hand held GPS or measured off adjacent drillholes. A compass was used to set up the rig azimuth and the dip was set using a clinometer. The final collar position was surveyed by a contract surveyor using differential GPS.

3.6.7 Density

There is no density data prior to 2010. Initially, density measurements were recorded using physical methods with vernier callipers to determine the length and diameter (volume) of a sample of square cut core and a digital balance used to determine the mass. Approximately 370 density measurements were taken using this method. In December 2011, utilisation of the water immersion technique commenced using diamond drill core and approximately 2,700 density measurements have been collected. Density is determined by weighing individual pieces of core that represent the sampled interval. A subjective decision as to the process used for the determination of the rock density is made by the geologist logging the hole. This depends on the weathering, fracturing and apparent porosity. Density measurements were taken at approximately 5 – 10 m intervals downhole in the mineralised zone. Certified weights were used to calibrate the scale daily and since most of the core was fresh and unweathered there was no need for wax coating to be applied prior to immersion. Both the calliper and immersion methods show the average density within the mineralised zones to be 2.89 t/m^3.

3.7 Data Verification

A geological borehole information system (GBIS) interface is used to access the SQL database. The validity of the database processes used for the Mineral Resource estimate of mineralisation at the Achmmach tin deposit has been confirmed via checks for internal consistency and robustness. These checks demonstrate that the drill hole data has been adequately validated with satisfactory QA-QC analysis and is appropriate for use in the estimation of Measured, Indicated and Inferred Mineral Resources which are included in this report.

3.8 Resource Estimation

Two separate resource estimates have been compiled, one for the Meknès Trend and the other for the Western Zone of the Sidi Addi Trend (Figure 3-26). The work was undertaken by Quantitative Group and their report is in Appendix 3A.

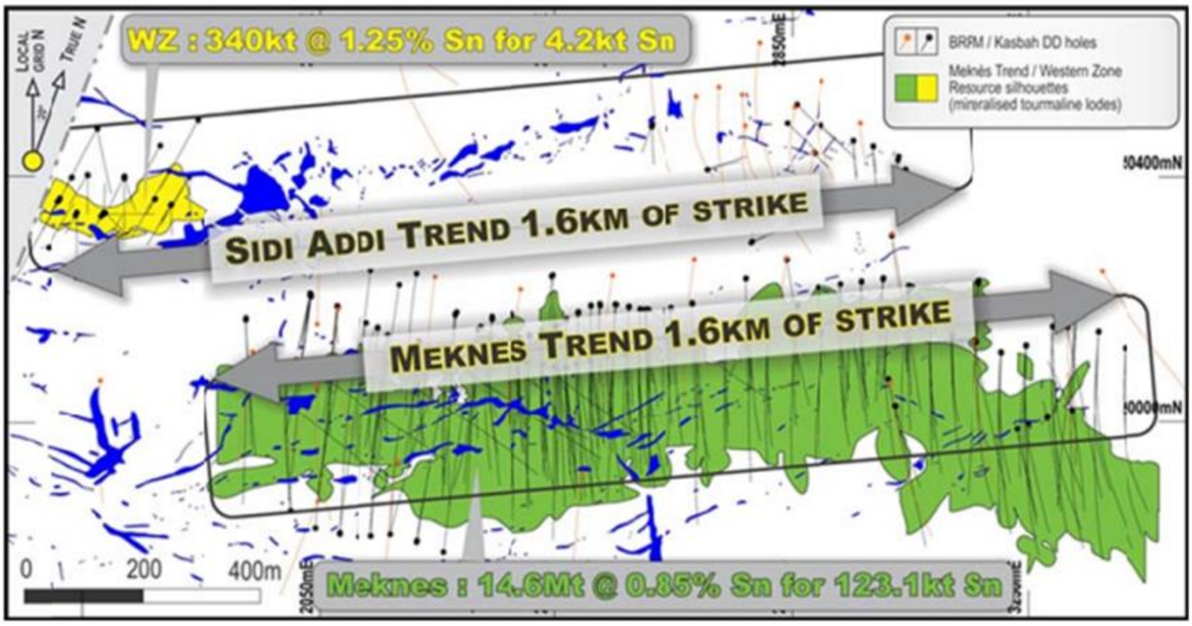


Figure 3-26 Location of Sidi Addi and Meknès Trends

3.9 Resource Estimation-Meknès Trend

3.9.1 Domaining

Improvements to domaining techniques with progressive resource iterations has led to greater statistical homogeneity, producing a resource more amenable to linear estimate techniques such as ordinary kriging (OK). Initially domains were based on potassium grades (as a proxy for tourmaline alteration e.g., a K_2O concentration below 0.26% indicates intense alteration), however more recent work has also included Sn grades (a 0.2% Sn bottom cut was used to assist in delineating outlines) in the geological interpretation. This effectively means the unmineralised tourmaline breccias are excluded from the wireframes.

Any mineralised material with good continuity that was not captured by the tin-tourmaline wireframing (several small continuous splays) was wireframed separately and included in the appropriate domain. Table 3-5 shows the numerical domain codes for the tin-tourmaline alteration zones.

Table 3-5 Meknès Trend domain codes

Domain number	Domain Name
0	Background
10	Vertical Feeders
30	Fez
40	Meknès
50	Marrakesh
60	East Zone Deeps
61	East Zone Shallow

Domain boundaries are defined as soft (grade interpolation of a block near the boundary is permitted from adjacent domains) or hard (grade interpolation of a block near the boundary is permitted only from within the domain from which it belongs). Domain boundary analysis for the Meknès Trend showed hard boundaries for all of the tin-tourmaline alteration domain variables.

3.9.2 Compositing

Drillhole data and wireframes were imported into Datamine for analysis. Composites were generated on 1 m downhole intervals for the purposes of resource estimation. Analysis of raw sample data showed that most sample lengths are 1 m. There were a few instances where uncomposited intervals were greater than 1 m. Since this was only 56 samples (of 2 m lengths from within the mineralized zones) of the 22,416 samples, the impact of re-compositing these to 1 m was considered negligible.

Some sections within tin-tourmaline altered domains had not been sampled. This is because only samples from the sections of drillhole core that register as Sn bearing on the Niton hand-held XRF analyser during the sampling process were sent for assay. It is incorrect to label these as null and to leave them blank as it is possible for a high grade being estimated in these areas. Therefore, a background sample grade of 0.005% Sn was applied. Table 3-6 shows the comparison of *Sn grade x sample length* before and after compositing and demonstrates there was no significant loss of data during the compositing process.

Table 3-6 Sn Accumulations before and after compositing

	TORDOM	Length	Sn Accm	Sn_Pct (Mean)
Raw	10	4,229	1,972	0.470
Composite	10	4,226	1,971	0.466
Diff %		99.9%	100.0%	99.3%
Raw	30	2,195	1,233	0.559
Composite	30	2,193	1,231	0.563
Diff %		99.9%	99.8%	100.7%
Raw	40	3,672	2,559	0.704
Composite	40	3,670	2,557	0.697
Diff %		99.9%	99.9%	99.1%
Raw	50	241	126	0.520
Composite	50	241	126	0.522
Diff %		99.9%	100.0%	100.2%
Raw	60	2,238	1,314	0.587
Composite	60	2,238	1,321	0.590
Diff %		100.0%	100.5%	100.7%
Raw	61	760	517	0.714
Composite	61	758	515	0.673
Diff %		99.8%	99.7%	94.3%

Bulk density data was not composited as samples were approximately 10 cm – 15 cm lengths of core and to composite these to 1 m intervals would be meaningless i.e. they are essentially point values.

3.9.3 Univariate Statistics

Summary statistics for all domains are shown in Table 3-7. Histograms for Meknès and East Zone Deeps are shown below (Figure 3-27). Sn is positively skewed and has some large values. Statistical top cuts were therefore applied. Table 3-7 shows the statistics for top cuts. Estimates were run for Sn with and without top cuts. Some data values for bulk density were outside acceptable ranges for this geological environment. Values above 3.5 t/m^3 or below 2 t/m^3 were rejected.

Table 3-7 Univariate data statistics

Domain	Variable	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Vertical Feeders	Sn_Pct	4245	0.00	13.65	0.47	0.81	0.66	1.72
	K_Pct	3706	0.01	5.35	1.17	1.23	1.51	1.05
	S_Pct	3693	0.01	8.97	0.90	0.85	0.72	0.94
	Density	882	1.26	4.17	2.90	0.11	0.01	0.04
Fez	Sn_Pct	2210	0.00	10.95	0.56	0.89	0.79	1.59
	K_Pct	1944	0.01	4.85	1.34	1.10	1.21	0.82
	S_Pct	1906	0.00	4.95	0.69	0.71	0.50	1.03
	Density	285	2.38	3.28	2.87	0.10	0.01	0.03
Meknès	Sn_Pct	3703	0.00	13.78	0.70	0.91	0.82	1.30
	K_Pct	3443	0.01	5.77	0.99	1.13	1.27	1.14
	S_Pct	3390	0.01	6.75	0.71	0.75	0.56	1.06
	Density	1059	1.38	4.24	2.89	0.13	0.02	0.04
Marrakesh	Sn_Pct	241	0.02	7.22	0.52	0.86	0.74	1.65
	K_Pct	241	0.01	4.36	0.74	0.91	0.83	1.23
	S_Pct	217	0.03	6.78	1.32	0.81	0.66	0.61
	Density	46	2.80	3.11	2.92	0.05	0.00	0.02
East Zone Deeps	Sn_Pct	2262	0.00	11.90	0.59	0.78	0.61	1.32
	K_Pct	1951	0.02	4.68	1.27	1.17	1.38	0.92
	S_Pct	1951	0.01	7.64	1.11	1.08	1.16	0.97
	Density	569	1.59	5.70	2.90	0.20	0.04	0.07
East Zone Shallow	Sn_Pct	782	0.00	11.35	0.67	1.14	1.29	1.70
	K_Pct	688	0.04	3.99	1.92	1.09	1.19	0.57
	S_Pct	688	0.00	9.70	1.53	1.31	1.70	0.86
	Density	103	2.36	3.44	2.88	0.13	0.02	0.05
Background	Sn_Pct	87314	0.00	2.46	0.05	0.10	0.01	2.00
	K_Pct	34271	0.01	5.64	2.51	1.12	1.26	0.45
	S_Pct	34207	0.00	8.82	1.02	0.78	0.61	0.76
	Density	2498	1.20	4.70	2.82	0.17	0.03	0.06

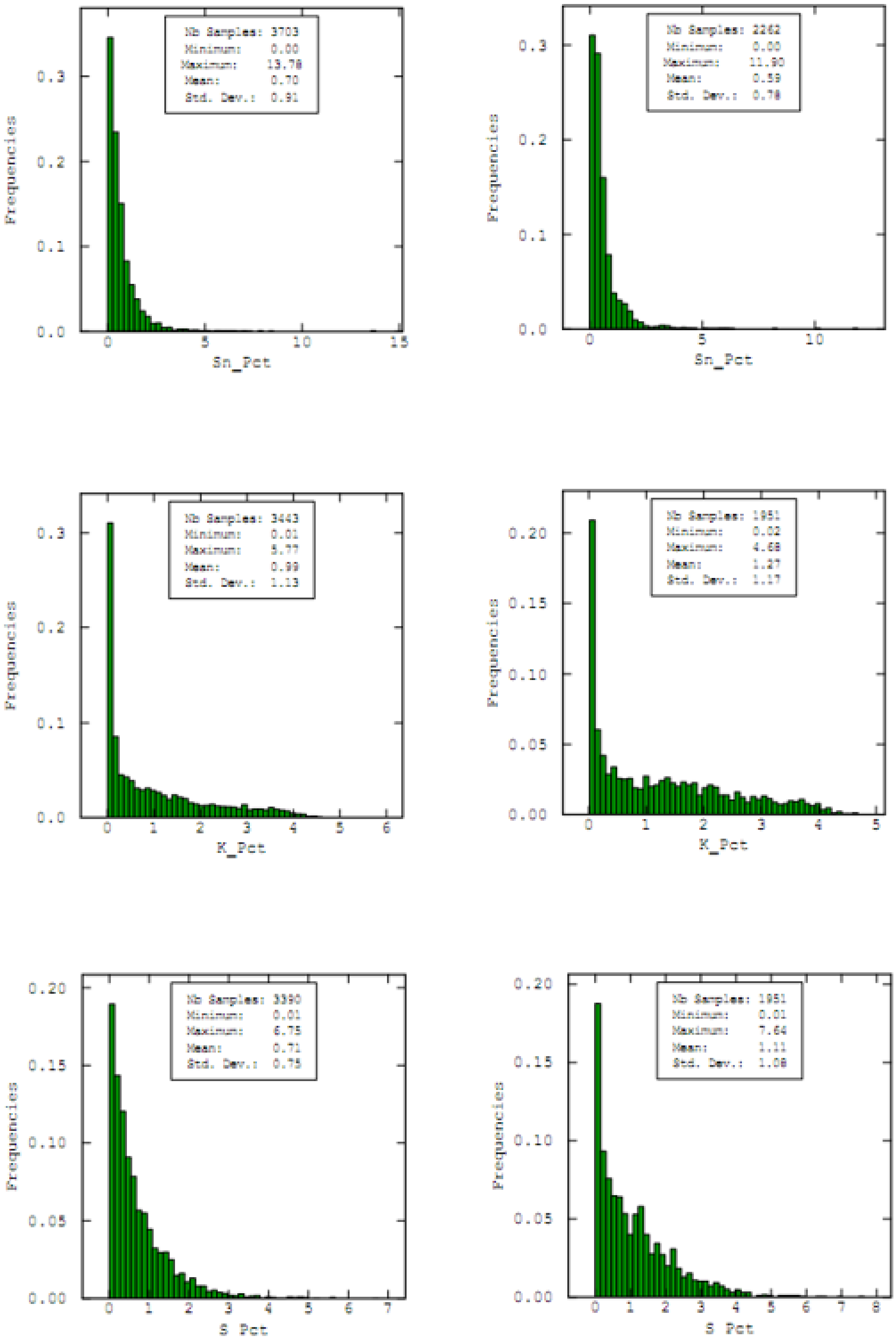


Figure 3-27 Histograms for Sn, K and S, Meknès (left) and East Zone Deeps (right)

Table 3-8 Statistics of top-cuts Sn

Zone	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Vertical Feeders	4,245	0	6.5	0.46	0.75	0.56	1.63
Fez	2,210	0	5.0	0.55	0.78	0.61	1.42
Meknès	3,703	0	6.5	0.69	0.85	0.72	1.23
Marrakesh	241	0.02	3.0	0.47	0.56	0.31	1.19
East Zone	2,262	0	5.0	0.58	0.70	0.49	1.21
East Zone Shallow	782	0	5.0	0.64	0.92	0.84	1.44

3.9.4 Declustering

As mentioned previously (Section 3.9.2) the higher-grade parts of mineralisation in the Meknès Trend were preferentially sampled. This could skew the estimate of the true grade for a given domain. Moving window cell declustering tests were performed for Sn in all domains and found that the plotted data stabilises at Step 5 (Figure 3-28) so a 200 m x 100 m x 6 m declustering grid was chosen. Statistics for the declustered variables are shown in Table 3-9.

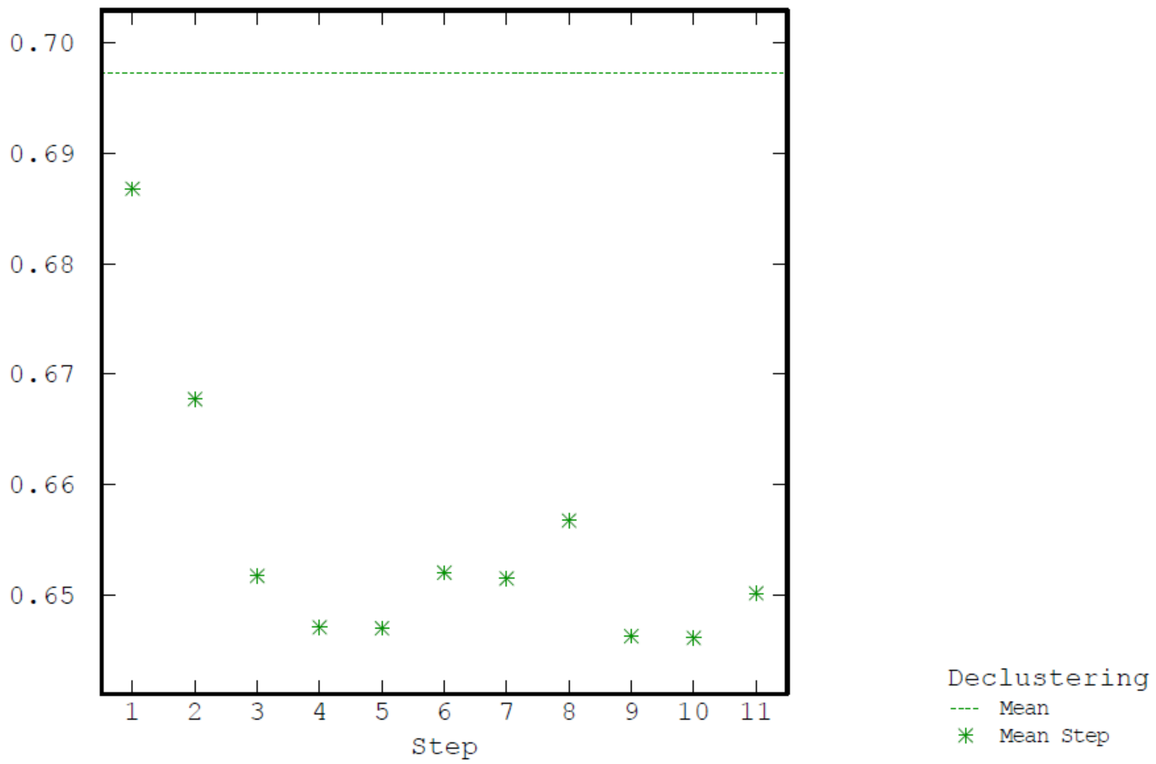


Figure 3-28 Grid declustering for Meknès Zone

Table 3-9 Unweighted vs. declustered statistics

	Tin		Potassium		Sulphur	
Domain	Raw Mean	Decl. Mean	Raw Mean	Decl. Mean	Raw Mean	Decl. Mean
Vertical Feeders	0.47	0.37	1.17	1.33	0.90	1.03
Fez	0.56	0.56	1.34	1.28	0.69	0.67
Meknès	0.70	0.65	0.99	1.06	0.71	0.76
Marrakesh	0.52	0.55	0.74	0.86	1.32	1.33
East Zone	0.59	0.65	1.27	1.29	1.11	1.08
East Zone Shallow	0.67	0.69	1.92	1.82	1.53	1.58

3.9.5 Variography

Variography was performed separately on Sn, K and S for each domain except the Marrakesh domain which had too few samples to produce meaningful variograms and so the Meknès models were used for this domain. Variograms for Sn in each domain were generated using data transformed into the Gaussian variable and then back transforming the model into the original data space, except for the West Zone Shallow. The Gaussian variograms are shown at the top of the following figures, and the back-transformed models at the bottom of the figures (Figure 3-29 to Figure 3-33).

Downhole experimental variograms were calculated using a lag of 1 m. These were used to determine the nugget effect. All experimental variograms were modelling using the nugget effect defined by the downhole variograms and two nested spherical models. Variograms for Sn are shown below. The red lines are the major direction, green lines the semi major and magenta lines the minor direction.

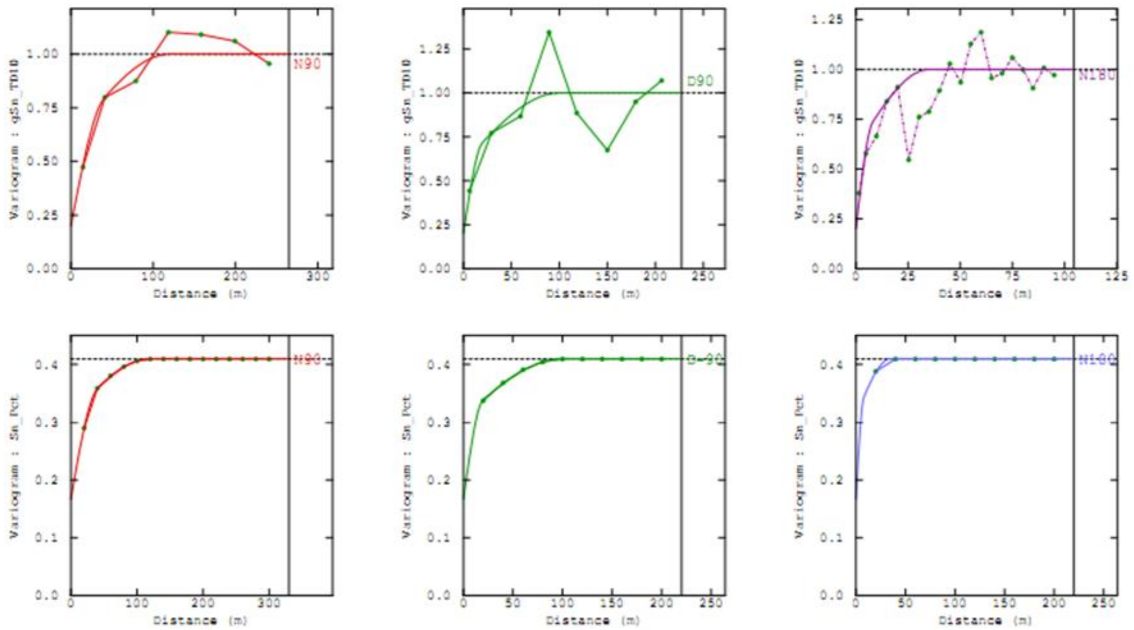


Figure 3-29 Vertical feeder variography

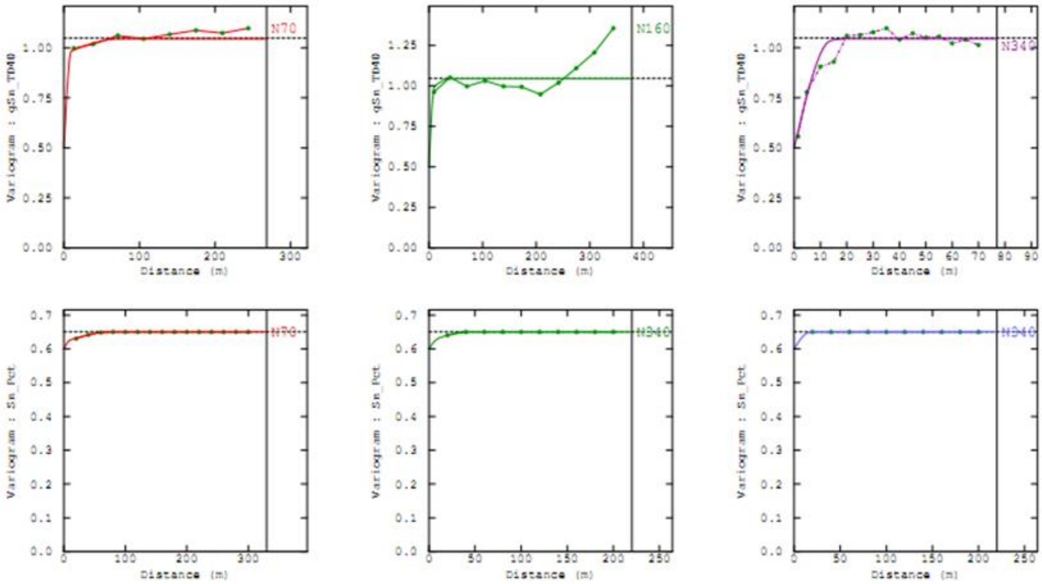


Figure 3-30 Meknès variography

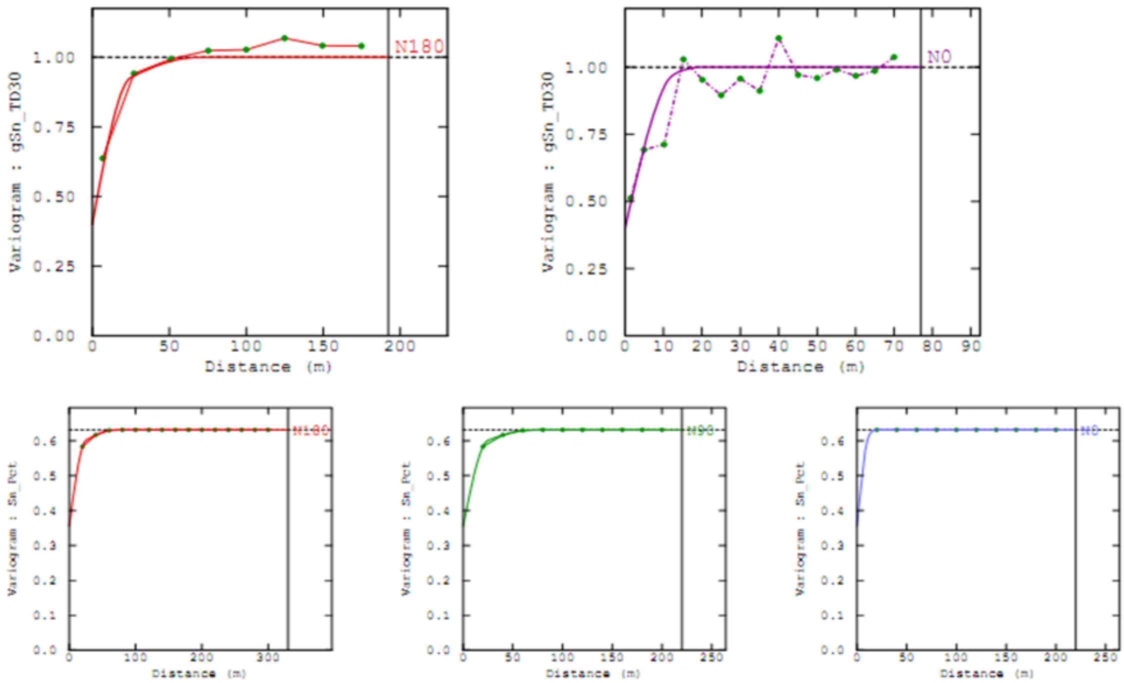


Figure 3-31 Fez variography

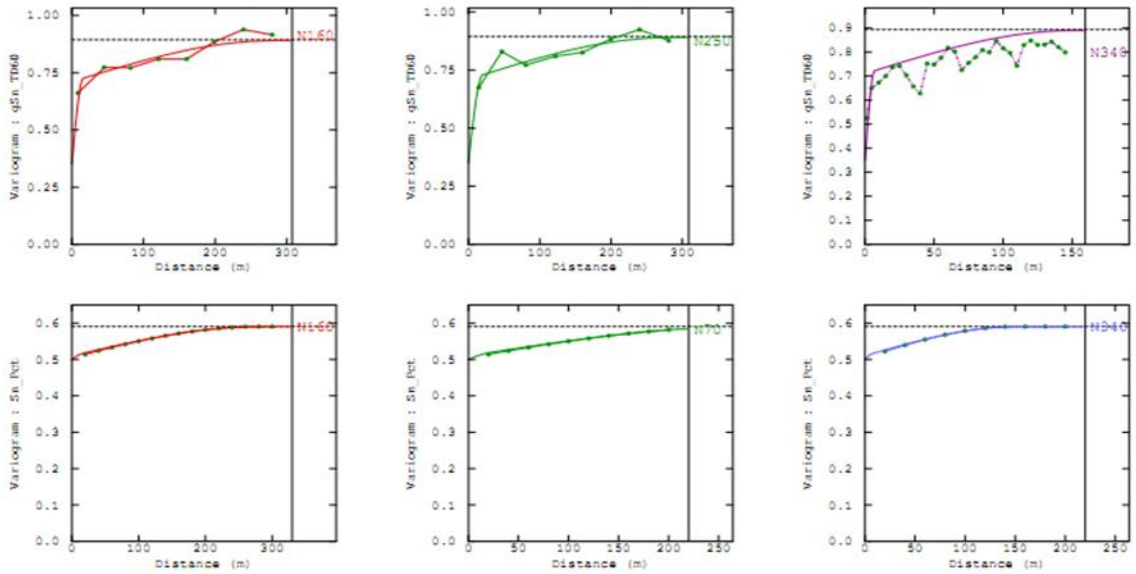


Figure 3-32 East Zone Deeps variography

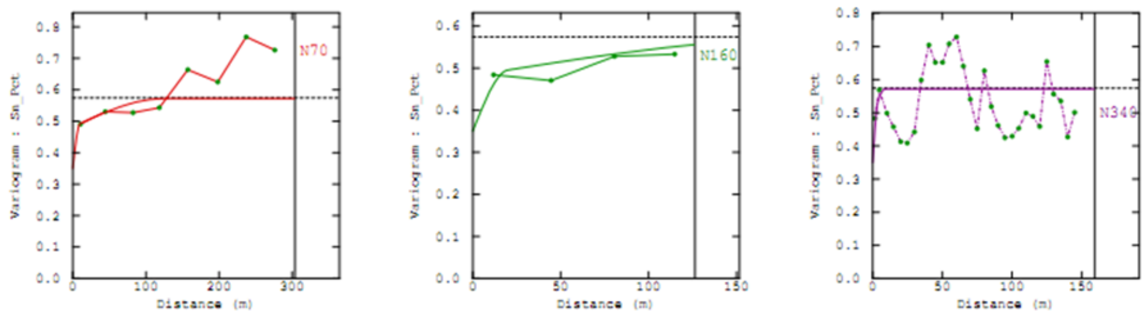


Figure 3-33 East Zone Shallow (non-transformed) variography

Model parameters for Sn, K and S are shown in Table 3-10, Table 3-11 and Table 3-12 respectively.

Table 3-10 Model Parameters for Sn

% Sn Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,2,1)			Major	Semi	Minor		(as %)	
Vertical Feeders	0,0,90	0,0,-90	0.167	40.70%	40	20	8	0.142	34.60%	1
					120	100	35	0.101	24.60%	2
Fez	-90,-30,0	90,30,0	0.355	56.20%	25	25	13	0.215	34.00%	1
					70	70	20	0.062	9.80%	2
Meknès	20,0,-30	-20,0,30	0.6	92.30%	10	10	14	0.02	3.10%	1
					75	40	20	0.03	4.60%	2
East Zone Deeps	-70,-40,0	70,40,0	0.5	84.70%	16	20	7	0.01	1.70%	1
					280	280	150	0.08	13.60%	2
		70,40,0	0.35	61.20%	20	10	4	0.13	23.10%	1

East Zone Shallow	-70,-40,0				200	125	9	0.09	15.70%	2
Background	20,0,-60	-20,0,60	0.008	45.50%	140	40	25	0.0045	25.60%	1
					550	150	210	0.0051	29.00%	2

Table 3-11 Model Parameters for K

K % Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,2,1)			Major	Semi	Minor		(as %)	
Vertical Feeders	0,0,90	0,0,-90	0.23	15.20%	5	3	3	0.5	33.10%	1
					50	35	20	0.78	51.70%	2
Fez	0,0,-50	0,0,50	0.4	33.10%	9	9	7	0.65	53.70%	1
					35	35	15	0.16	13.20%	2
Meknès	0,0,-60	0,0,60	0.4	31.50%	10	10	10	0.55	43.30%	1
					130	130	20	0.32	25.20%	2
East Zone	-90,-40,0	90,40,0	0.3	21.70%	20	35	6	0.66	47.80%	1
					165	100	30	0.42	30.40%	2
East Zone Shallow	-90,-40,0	90,40,0	0.44	36.70%	60	30	6	0.5	41.70%	1
					120	100	70	0.26	21.70%	2
Background	0,0,-60	0,0,60	0.4	36.40%	14	14	8	0.44	40.00%	1
					90	90	80	0.26	23.60%	2

Table 3-12 Model Parameters for S

S % Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,2,1)			Major	Semi	Minor		(as %)	
Vertical Feeders	0,0,90	0,0,-90	0.17	23.80%	35	150	15	0.18	25.20%	1
					500	175	20	0.365	51.00%	2
Fez	0,0,-50	0,0,50	0.1	20.10%	10	10	7	0.14	28.10%	1
					100	100	40	0.258	51.80%	2
Meknès	-10,0,-50	10,0,50	0.1	17.70%	25	5	6	0.22	38.90%	1
					80	60	25	0.245	43.40%	2
East Zone	0,0,-40	0,0,40	0.25	21.30%	20	20	10	0.53	45.10%	1
					380	380	60	0.4	33.60%	2
East Zone Shallow	0,0,-40	0,0,40	0.58	37.20%	40	30	9	0.44	28.20%	1
					150	100	16	0.54	34.60%	2
Background	0,0,-60	0,0,60	0.2	32.30%	50	40	20	0.28	45.20%	1
					440	150	250	0.14	22.60%	2

3.9.6 Bulk Density Variography

Variograms for bulk density were generated for all mineralised domains (Figure 3-34). Model and model parameters are shown below (Table 3-13).

Table 3-13 Model Parameters for Bulk Density

Density Variogram Model											
Domain	Isatis	Datamine	Nugget	Nugget	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,2,1)	(C0)	(as %)	Major	Semi	Minor		(as %)	
Mineralised Zones	0,0,-30	0,0,30	0.0045	43.6%	15	15	14	0.0014	13.2%	1
					60	60	30	0.0045	43.2%	2
Background	0,0,-30	0,0,30	0.008	37.0%	60	60	30	0.007	32.4%	1
					600	600	40	0.0066	30.6%	2

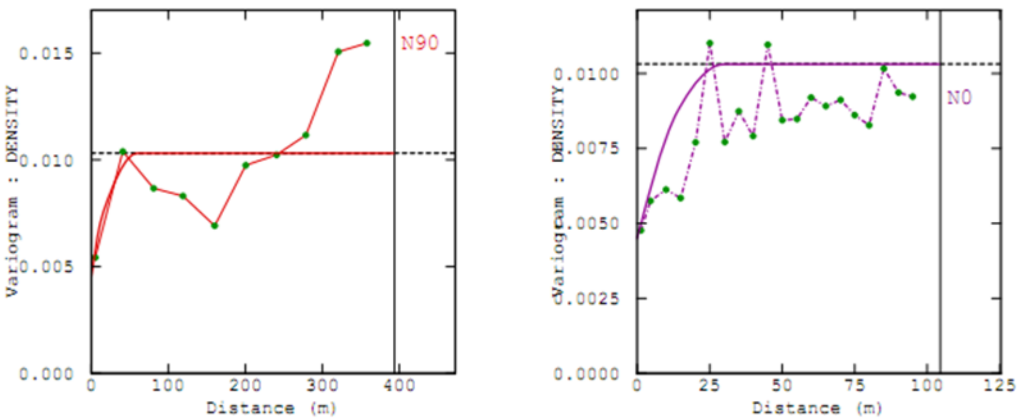


Figure 3-34 Bulk density variography

3.9.7 OK Estimation

Model

A volume block model was constructed in Datamine using the details in Table 3-14. Datamine convention has the origin as the lowermost south west corner of the first block. The horizontal parent block size is approximately half of the drill spacing for most of the deposit, except for the centre of the Meknès Zone where the drill spacing is closer. Block model volumes per domain were compared with wireframe volumes to check for consistence and were satisfactory.

Table 3-14 Block model details

	Model Origin	Block Size (m)	Number of Blocks	Model Limit	Min. sub-cell
Easting (X)	1,800	20	90	3,600	5
Northing (Y)	49,700	20	35	50,400	2.5
RL (Z)	500	5	150	1,250	2.5

Neighbourhood Analysis

Quantitative kriging neighbourhood analysis was conducted to optimise search criteria. Ten test estimates were run in the Meknès Zone that used 5, 10, 15, 20, 25, 30, 40, 60, 100, 200 as the maximum number of samples. Results for the >30 showed over smoothing and < 15 were patchy. Therefore, the maximum number of samples chosen was in between these two values. The following tables summarise the chosen parameters of the search neighbourhoods for Sn and density (Table 3-15 and Table 3-16).

Table 3-15 Sn search parameters

% Sn Search Parameters								
Domain	Major	Range Semi	Minor	No. of Samples		Octants Used?	Max. Samp per hole	Disc.
				Min.	Max.			
Vertical Feeders	100	80	30	5	15	N	N/A	5x5x3
Fez	70	70	20	5	15	N	N/A	5x5x3
Meknès	75	40	20	5	15	N	N/A	5x5x3
East Zone	200	200	125	5	15	N	N/A	5x5x3
East Zone Shallow	150	100	10	5	15	N	N/A	5x5x3
Background	50	50	25	5	15	N	N/A	5x5x3

Table 3-16 Density search parameters

Density t/m3 Search Parameters								
Domain		Range		No. of Samples		Octants Used?	Max. Samp per hole	Disc.
	Major	Semi	Minor	Min.	Max.			
Minl. Zones	60	60	30	10	20	N	N/A	5x5x3
Background	300	300	40	10	20	N	N/A	5x5x3

Dynamic Anisotropy

A dynamic search feature within Datamine allows the geometry of the search ellipse to be defined separately for each block. This is an advantage where the average dip and dip direction does not honour the local grade geometry. The wireframe triangles were used to estimate local dip and dip direction (user set tolerances mitigate the risk of generating invalid points), and these points were then used to define the geometry of the mineralized structures. These were treated as variables and during the estimation process, the search ellipse and variogram orientations were rotated individually for each block.

Estimation

Estimation was completed using ordinary kriging utilising the parameters defined in the sections above. A second estimation pass utilising a larger search ellipse (twice the initial search distance) was run if blocks remained unfilled after the first pass. Table 3-17 shows the blocks estimate by each pass.

Table 3-17 Search passes used for Sn

Pass Number	1		2		Not Estimated	
	BCM	% Filled	BCM	% Filled	BCM	% Filled
Vertical Feeders	5,289,125	91%	521,875	9%	15,438	0%
Fez	797,594	100%	2,156	0%	0	0%
Meknès	2,343,719	99%	33,094	1%	0	0%
Marrakesh	428,344	98%	10,627	2%	125	0%
East Zone Deeps	2,960,000	100%	0	0%	0	0%
East Zone Shallow	457,156	100%	312	0%	0	0%
Total	12,275,938	95%	568,063	4%	15,563	0%

Most blocks were filled on the first pass. The large number of vertical feeder blocks unfilled by the first pass was due to these extending at depth away from drilling. Blocks not filled on the second pass were assigned default values of 0.3% Sn for Vertical Feeders and 0.2%Sn for Marrakesh. Default values of 1% K and 0.7%S were also applied in unfilled blocks in the Vertical Feeders domain.

A large number of blocks, distal to drilling, in the background domain remained un-estimated. These were assigned values according to known mineralogical occurrences. (I.e. 0.005% Sn, 2.5% K and 0.2% S above the 950mRL and 0.7% S below it). Default values for bulk density included 2.85t/m^3 for the vertical feeders and 2.80t/m^3 for the background.

Validation

Model validation included performing a nearest neighbour estimate in additional to kriging to identify any bias, comparing model grade to sample grades spatially (i.e., on-screen in 3D) and statistical analysis of composite data compared to model data (see Table 3-18 below). All estimates behaved as expected (kriging produced smoother results and no bias was detected) and validation demonstrates that the estimate was satisfactory. Swath plots for the Meknès Zone domain are included below. Swath plots for other domains are seen in Figure 3-35 and Figure 3-36.

Table 3-18 Composite vs. model means for mineralized domains

Tin				
Domain	Raw Mean	Decl. Mean	Model	%Diff.
Vertical Feeders	0.47	0.37	0.37	99.8%
Fez	0.56	0.56	0.59	104.8%
Meknès	0.70	0.65	0.66	101.6%
Marrakesh	0.52	0.55	0.57	103.4%
East Zone Deeps	0.59	0.65	0.58	89.7%
East Zone Shallow	0.67	0.69	0.71	102.8%

Potassium				
Domain	Raw Mean	Decl. Mean	Model	%Diff.
Vertical Feeders	1.17	1.33	1.31	98.8%
Fez	1.34	1.28	1.34	104.3%
Meknès	0.99	1.06	0.95	89.3%
Marrakesh	0.74	0.86	0.79	91.3%
East Zone Deeps	1.27	1.29	1.22	94.8%
East Zone Shallow	1.92	1.82	1.82	100.2%

Sulphur				
Domain	Raw Mean	Decl. Mean	Model	%Diff.
Vertical Feeders	0.900	1.030	0.99	96.3%
Fez	0.690	0.670	0.73	109.6%
Meknès	0.710	0.760	0.77	101.0%
Marrakesh	1.320	1.330	1.45	109.4%
East Zone Deeps	1.110	1.080	1.13	104.3%
East Zone Shallow	1.530	1.580	1.58	99.8%

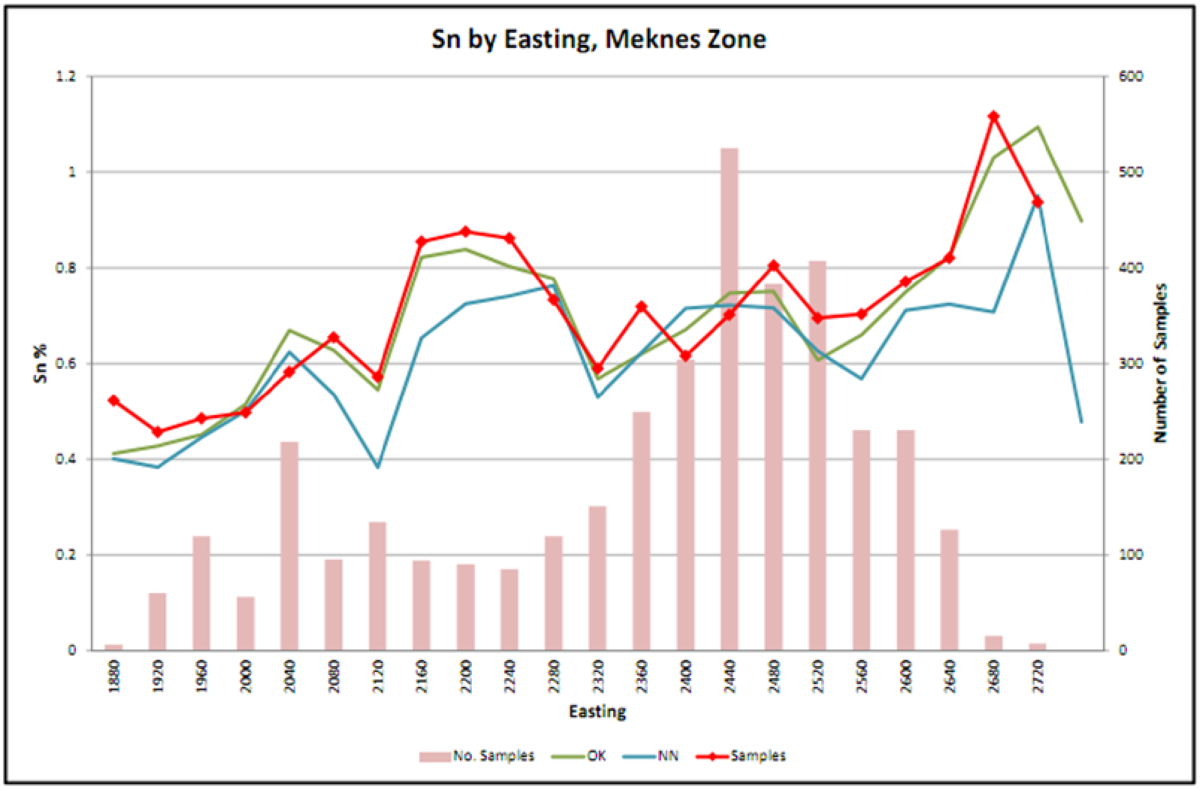


Figure 3-35 Swath Plot for Meknès Zone by Easting

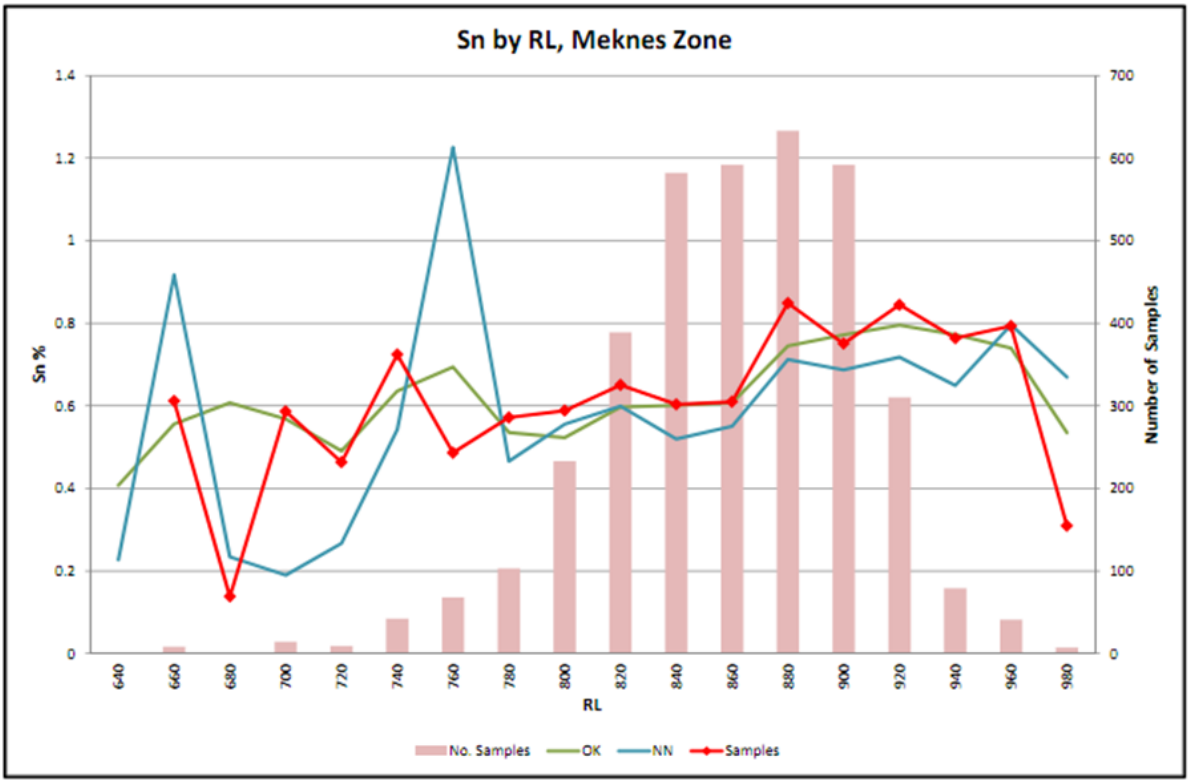


Figure 3-36 Swath Plot for Meknès Zone by RL

3.10 Resource Estimation-Western Zone (Sidi Addi Trend)

3.10.1 Domaining

A tin grade of approximately 0.4% Sn was used to generate the wireframes. The Main Zone dips to the NNW at 60° and the Branches average about 45° in the same direction. Three domains were recognized and are presented in Table 3-19.

Table 3-19 Western Zone domain codes

Domain Code (TORDOM)	Domain Name
0	Background
70	Main Zone
75	Branches

Analysis of domain boundaries showed that the use of hard boundaries is appropriate for all variables (Sn, K and S) for this model.

3.10.2 Compositing

Drillholes were composited to 1m intervals using Datamine software, with an allowable minimum composite of 0.25m. Table 3-20 shows that the tin accumulations before and after compositing are acceptable with no significant loss of metal content. As for the Meknès Trend, bulk density data for the Western Zone was not composited.

Table 3-20 Sn Accumulations before and after compositing

	Zone	Length	Sn Accm	% Sn (mean)
Raw	Main	137.3	181	1.49
Composite		136.7	180	1.36
Difference		99.6%	99.6%	91.3%
Raw	Branches	161.4	127	0.813
Composite		160.2	126	0.806
Difference		99.2%	99.2%	99.1%

3.10.3 Univariate Statistics

Summary statistics are shown below in Table 3-21 and histograms are shows in Figure 3-37. Tin is positively skewed with some high data values. Top-cuts were applied to Sn only. These were 5.5% Sn for the Main Zone and 1% Sn for the Branches. Bulk density data was reviewed and found to be within acceptable limits.

Table 3-21 Univariate Data Statistics

Domain	Variable	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Main Zone	% Sn	141	0.043	11.60	1.37	1.47	2.16	1.08
	% K	141	0.030	3.54	0.46	0.63	0.40	1.37
	% S	141	0.005	4.77	0.15	0.48	0.23	3.32
	Density	80	2.280	3.37	2.82	0.14	0.02	0.05
Branch Zone	% Sn	163	0.014	4.48	0.81	0.67	0.45	0.83
	% K	163	0.030	3.57	0.83	0.73	0.54	0.89
	% S	163	0.005	3.70	0.20	0.47	0.22	2.35
	Density	61	2.480	2.91	2.74	0.10	0.01	0.04
Background	% Sn	1553	0.001	1.18	0.11	0.12	0.01	1.09
	% K	1553	0.030	5.33	1.85	1.09	1.19	0.59
	% S	1553	0.005	7.55	0.44	0.68	0.47	1.57
	Density	193	2.260	3.36	2.71	0.15	0.02	0.05

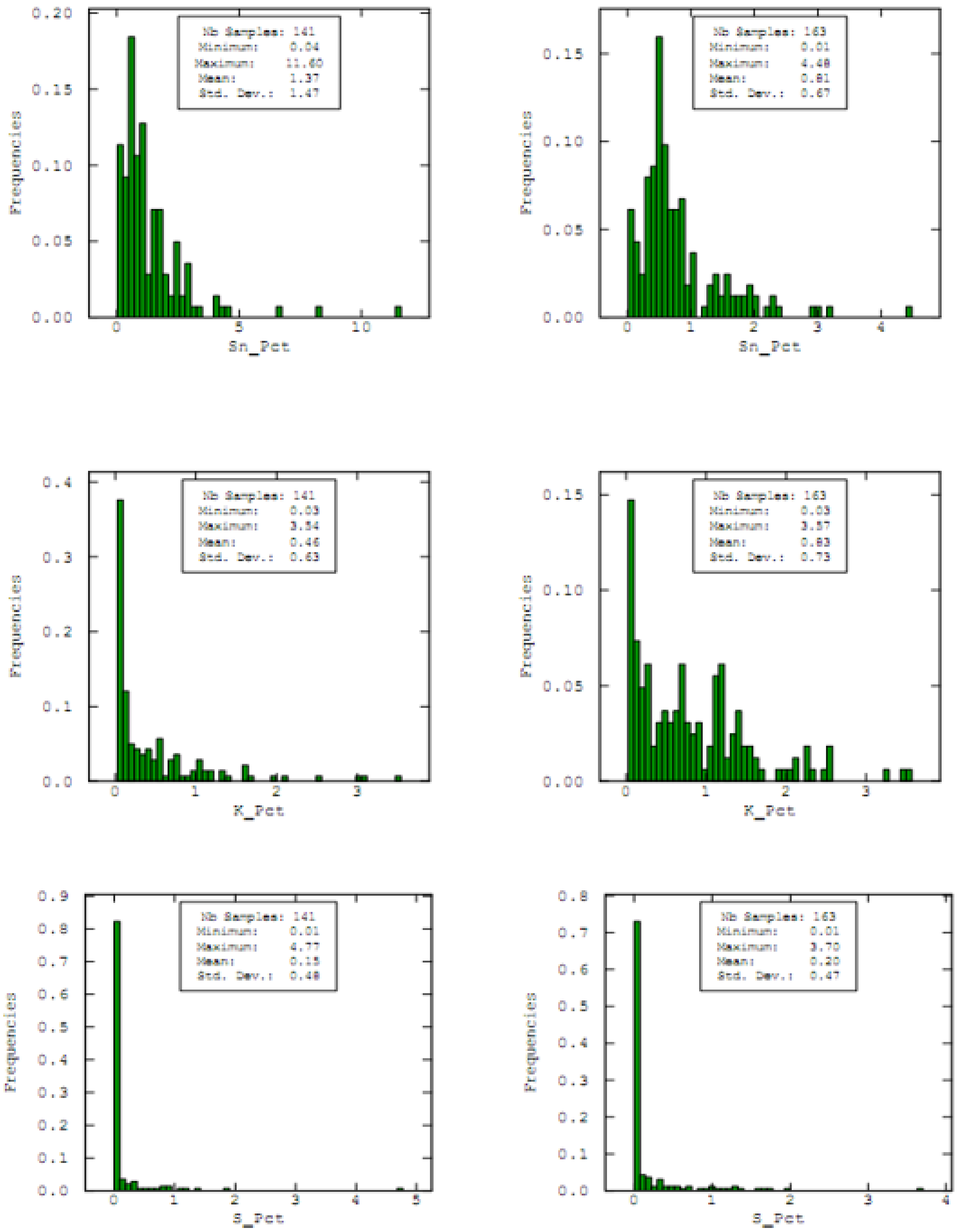


Figure 3-37 Histograms for Sn, K and S for Main Zone (left) and Branches (right)

3.10.4 Declustering

Moving window cell declustering tests were performed by Job (2014) using a 50 mx50 m x 2.5 m declustering grid for the Main Zone, a 40 m x 40 m x 10 m grid for the Branches. Declustered statistics are presented in Table 3-22.

Table 3-22 Declustered Statistics

Domain	Variable	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Main Zone	Sn_Pct	141	0.04	11.60	1.34	1.38	1.90	1.03
	K_Pct	141	0.03	3.54	0.45	0.61	0.37	1.34
	S_Pct	141	0.01	4.77	0.14	0.46	0.22	3.22
Branch Zone	Sn_Pct	163	0.014	4.48	0.90	0.67	0.44	0.74
	K_Pct	163	0.030	3.57	0.93	0.77	0.59	0.83
	S_Pct	163	0.005	3.70	0.26	0.52	0.27	2.21

3.10.5 Variography

Despite the small sample population, relatively good variograms could be constructed in the plane of mineralisation and across strike. Downhole variograms were generated and used to determine the nugget effect. Variograms and model parameters are included below (Figure 3-38 to Figure 3-40 and Table 3-23 to Table 3-25).

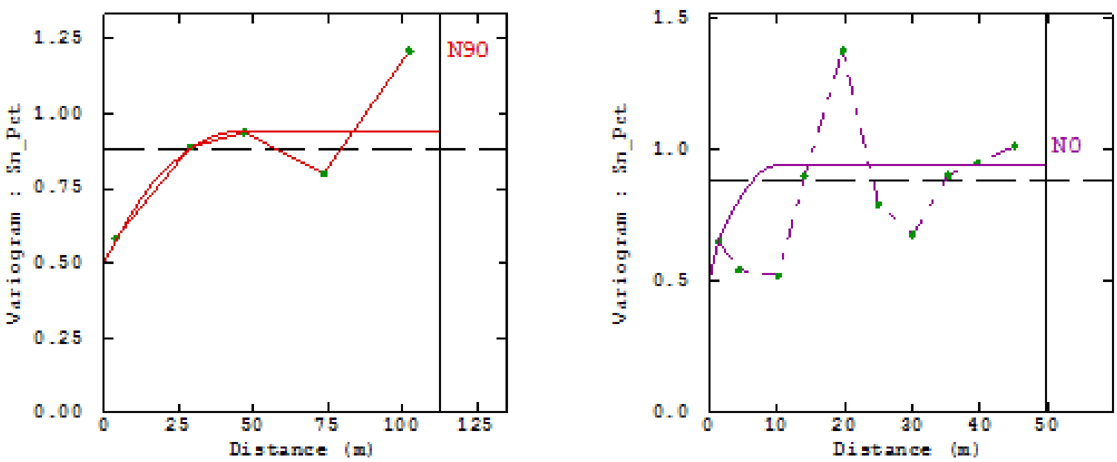


Figure 3-38 Main Zone Variography

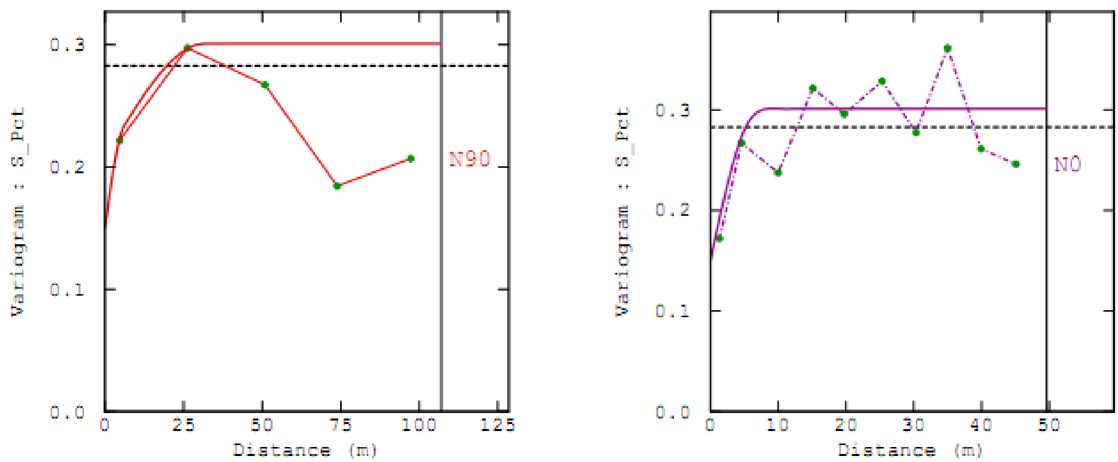


Figure 3-39 Branches Variography

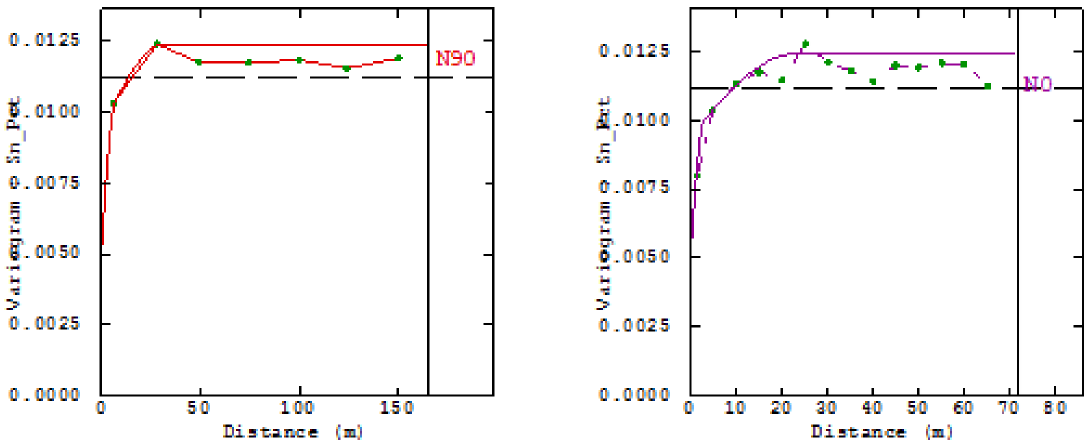


Figure 3-40 Background Variography

Table 3-23 Model Parameters for Sn

S % Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,1,3)			Major	Semi	Minor		(as %)	
Main Zone	0,0,-60	0,60,0	0.5	53.2%	20	20	2	0.09	9.60%	1
					45	45	10	0.35	37.20%	2
Branches	0,0,-45	0,45,0	0.58	37.2%	6	6	6	0.057	18.90%	1
					32	32	8	0.094	31.20%	2
Background	0,0,-60	0,60,0	0.005	40.3%	7	7	3	0.0044	35.50%	1
					32	32	23	0.003	24.20%	2

Table 3-24 Model Parameters for K

K % Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,1,3)			Major	Semi	Minor		(as %)	
Main Zone	0,0,-60	0,60,0	0.08	33.5%	5	5	3	0.124	51.9%	1
					55	55	8	0.035	14.6%	2
Branches	0,0,-45	0,45,0	0.07	20.1%	5	5	8	0.143	41.1%	1
					34	34	15	0.135	38.8%	2
Background	0,0,-60	0,60,0	0.25	22.3%	9	9	2	0.256	22.8%	1
					45	30	14	0.615	54.9%	2

Table 3-25 Model Parameters for S

% Sn Variogram Models										
Domain	Isatis	Datamine	Nugget (C0)	Nugget (as %)	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,1,3)			Major	Semi	Minor		(as %)	
Main Zone	0,0,-60	0,60,0	0.01	32.3%	5	5	2	0.0064	20.6%	1
					32	32	5	0.0146	47.1%	2
Branches	0,0,-45	0,45,0	0.015	17.6%	80	80	35	0.07	82.4%	1
										2
Background	0,0,-60	0,60,0	0.05	13.7%	9	9	2	0.038	10.4%	1
					185	185	75	0.278	76.0%	2

3.10.6 Bulk Density Variography

Bulk density data produced reasonable variograms shown below with the model parameters (Figure 3-41 and Table 3-26).

Table 3-26 Model parameters for bulk density

Density Variogram Model										
Domain	Isatis	Datamine	Nugget	Nugget	Range			Sill	Sill	Structure
	Math Rot.	Rot. (3,2,1)	(C0)	(as %)	Major	Semi	Minor		(as %)	
Mineralised Zones	0,0,-60	0,60,0	0.003	22.4%	13	13	12	0.0046	34.3%	1
					36	36	34	0.0058	43.3%	2
Background	0,0,-60	0,60,0	0.004	22.9%	15	15	15	0.0039	22.3%	1
					36	36	25	0.0096	54.9%	2

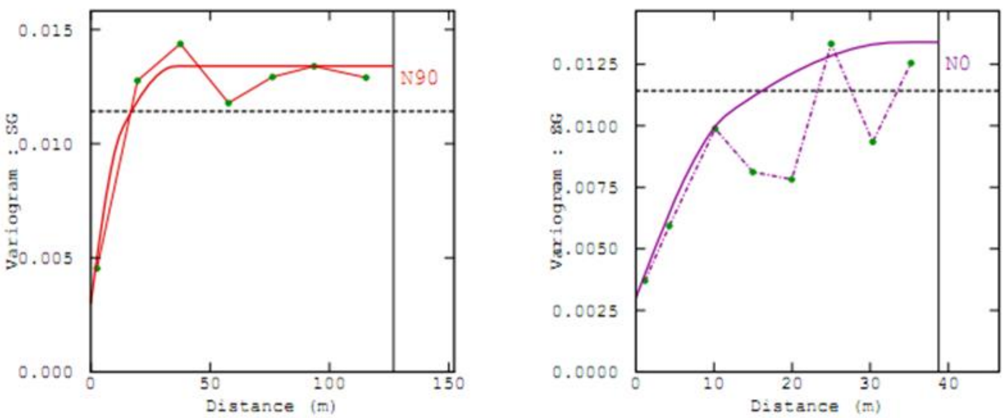


Figure 3-41 Bulk density variography

3.10.7 Ordinary Kriging Estimation

Model

A volume block model was generated in Datamine. Details are shown in Table 3-27. The horizontal parent block size is approximately half to one-third of the drill spacing. A comparison of wireframe versus block model volumes shows that the constructed block model is similar volumetrically.

Table 3-27 Block model details

	Model Origin	Block Size (m)	Number of Blocks	Model Limit	Min. sub-cell
Easting (X)	1,400	10	55	1,950	2
Northing (Y)	50,150	10	40	50,550	2
RL (Z)	800	5	80	1,200	1

Neighbourhood Analysis

Search parameters derived from Quantitative Kriging Neighbourhood Analysis (QKNA) for Sn and bulk density are shown in the tables below (Table 3-28 and Table 3-29).

Table 3-28 Sn search parameters

% Sn Search Parameters								
Domain		Range		No. of Samples		Octants Used?	Max. Samp per hole	Disc.
	Major	Semi	Minor	Min.	Max.			
Main Zone	40	40	10	8	20	N	N/A	5x5x5
Branches	30	30	7	8	20	N	N/A	5x5x5
Background	30	30	20	8	20	N	N/A	5x5x5

Table 3-29 Bulk density search parameters

Density t/m3 Search Parameters								
Domain		Range		No. of Samples		Octants Used?	Max. Samp per hole	Disc.
	Major	Semi	Minor	Min.	Max.			
Mineralised Zones	35	35	35	8	20	N	N/A	5x5x5
Background	35	35	25	8	20	N	N/A	5x5x5

Estimate

Estimation was completed using ordinary kriging utilising the parameters defined in the sections above. Consecutive estimation passes utilising larger search ellipses (2nd Pass was twice the initial search distance, 3rd pass was 4 times the search distance) were required as blocks remained un-estimated. Table 3-30 shows the block estimate percentages by each pass.

Table 3-30 Search passes for Sn

Pass Number	1		2		3	
	BCM	% Filled	BCM	% Filled	BCM	% Filled
Main Zone	81,380	91.10%	7,936	8.90%	28	0.0%
Branches	13,352	31.20%	26,188	61.10%	3,288	7.7%

Default values of 0.005% Sn and, 2% K and 0.1% S and 2.65t/m^3 for bulk density were applied to distal blocks in the Background domain that remained unfilled.

Validation

Block model validation included: performing a nearest neighbour estimate in additional to kriging to identify any bias; comparing model grade to sample grades spatially and statistical analysis of composite data compared to model data. All data behaved as expected (kriging produced smoother results and no bias was detected) and validation demonstrated that the estimate was satisfactory. Swath plots for Sn in both mineralized domains are included below (Figure 3-42 to Figure 3-45).

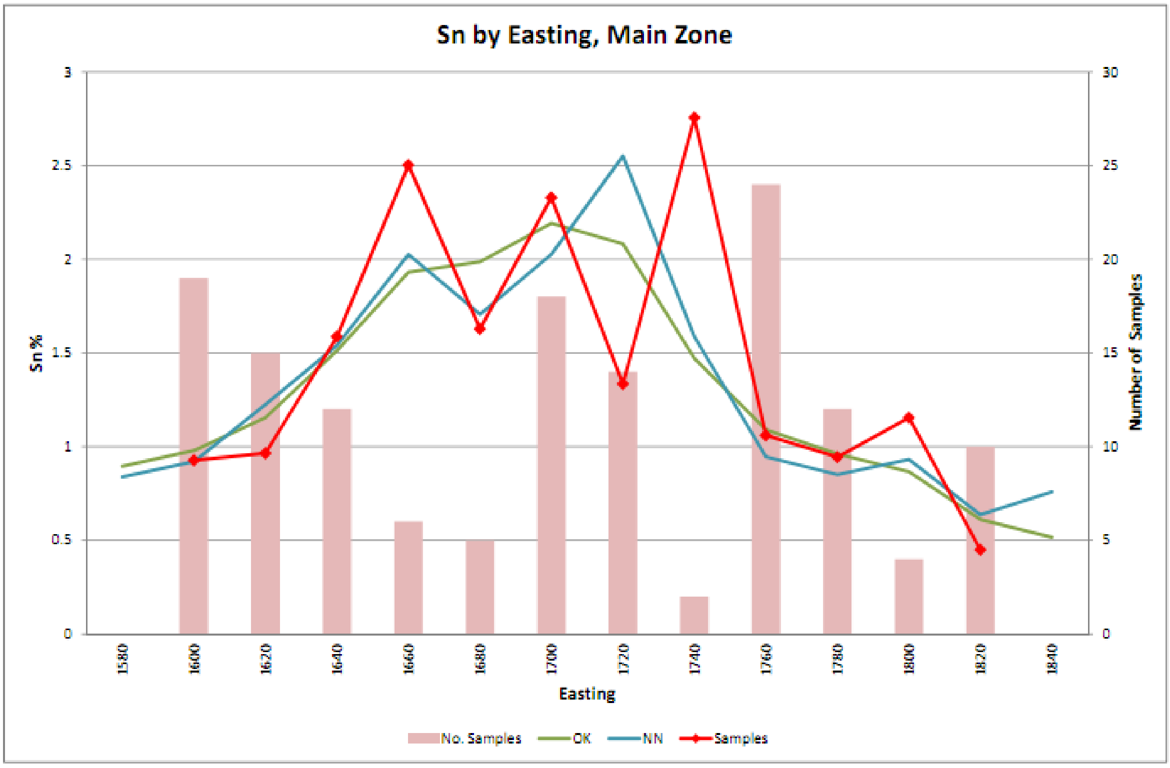


Figure 3-42 Swath plot main zone by easting

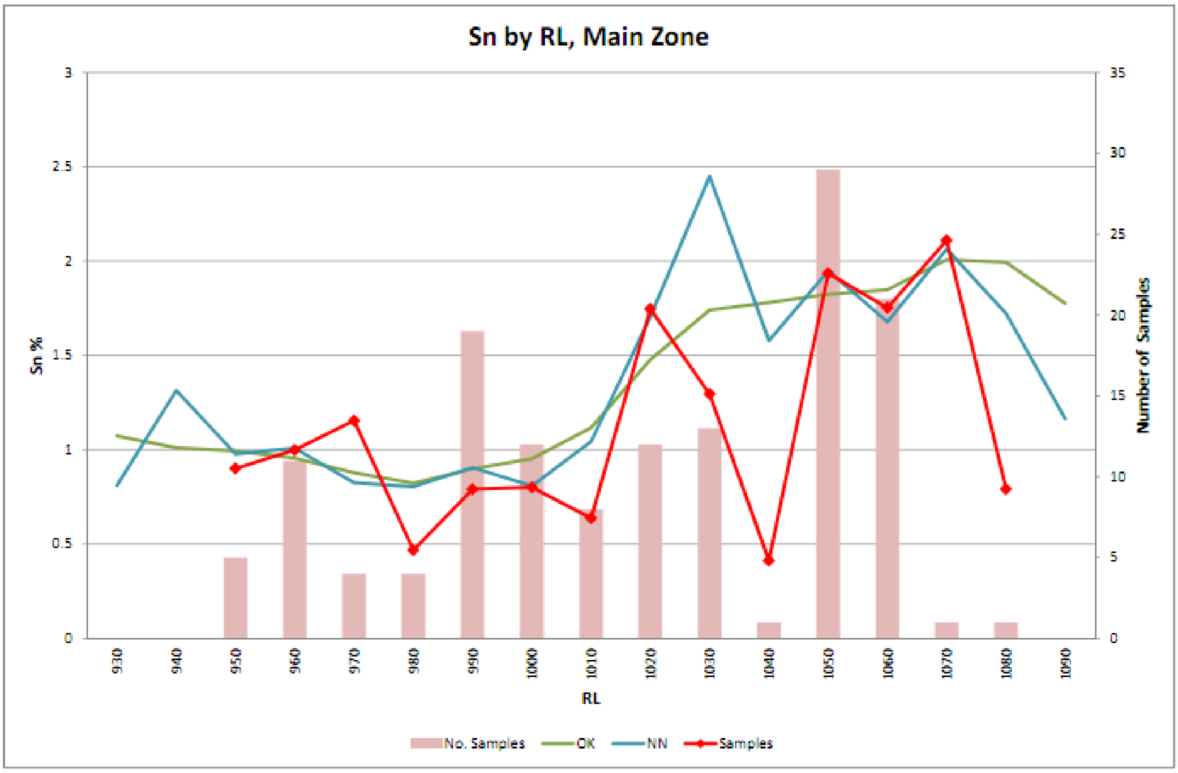


Figure 3-43 Swath plot main zone by RL

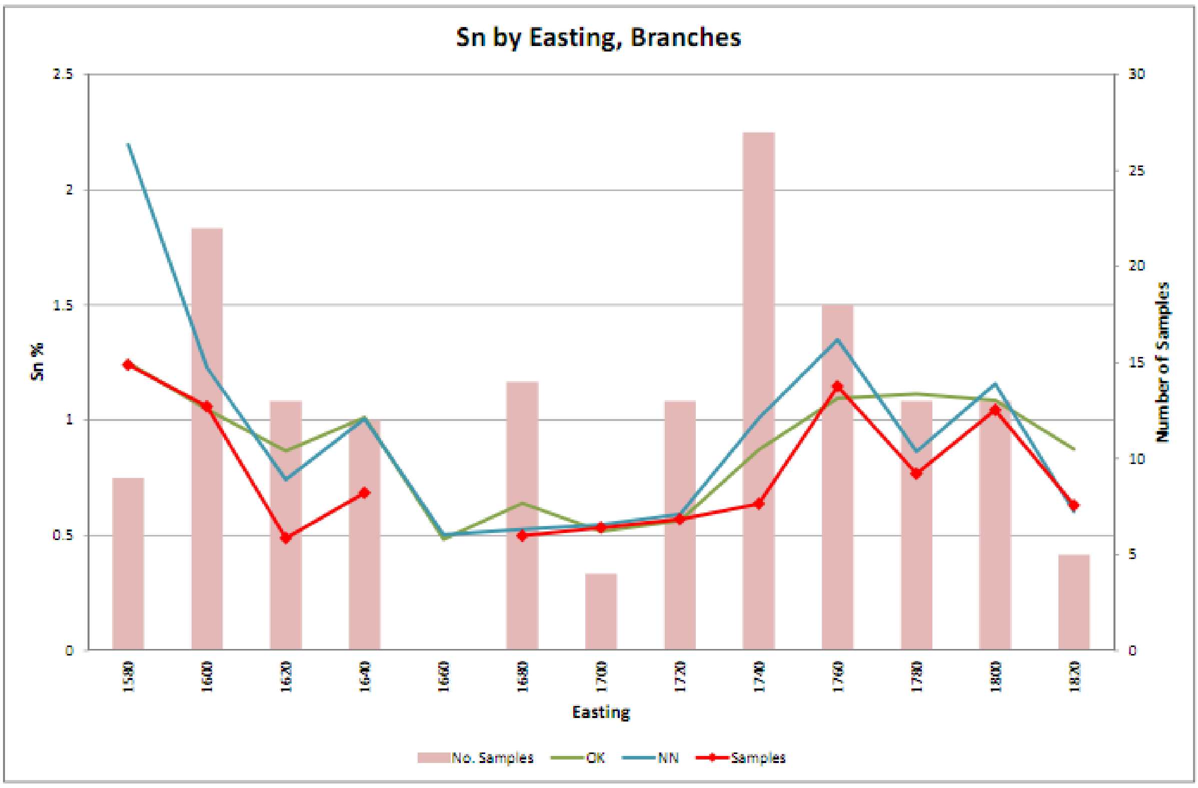


Figure 3-44 Swath plot branches by easting

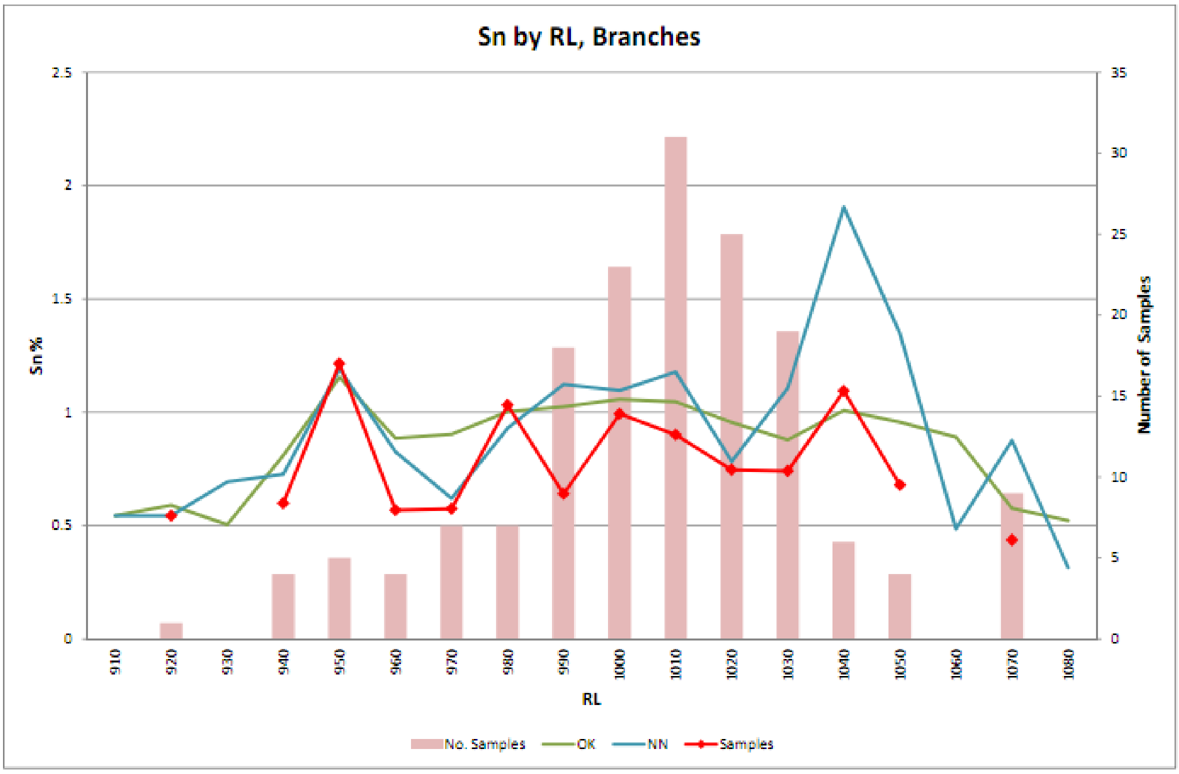


Figure 3-45 Swath plot branches by RL

3.11 Mineral Resource Classification

The classification of the resource estimate was based on the flowing parameters:

- data quality and quantity
- spatial continuity of Sn mineralisation
- geological interpretation and domaining
- data spacing of drill intersects
- previous classification of the resource estimate
- search pass data.

Diamond drill spacing ranges between 20 m – 40 m spacing. All areas were well sampled, logged and assayed. Domaining is appropriate and robust. Interpretations are not over-extrapolated (20 m up and down dip and 20 m – 40 m along strike depending on the drill spacing).

Therefore, areas in the core of the Meknès Zone and Vertical Feeders, where the drill spacing is 20 m and continuity of grade is consistent along strike, are classified Measured. The remainder of the deposit with drill spacing between 30 m – 40 m is classified as Indicated.

3.12 Mineral Resource Reporting

The mineral resource is reported to a cut-off grade of 0.5% Sn for both Meknès Trend and Western Zone.

3.12.1 Meknès Trend

The reported Mineral Resource at a cut-off is 0.5% Sn for the underground component for both Meknès Trend and Western Zone is shown in Table 3-31 below.

Table 3-31 Meknès Trend Mineral Resource

Classification	MTonnes	Sn%	Sn (kt)
Measured	1.6	1.00	16.1
Indicated	13.0	0.80	107.0
Inferred	-	-	-
Total	**14.6**	**0.85**	**123.1**

The Sn grade in both these tables has been rounded to the nearest 0.05% Sn.

The grade tonnage curve for all reportable resource categories is shown below (Figure 3-46).

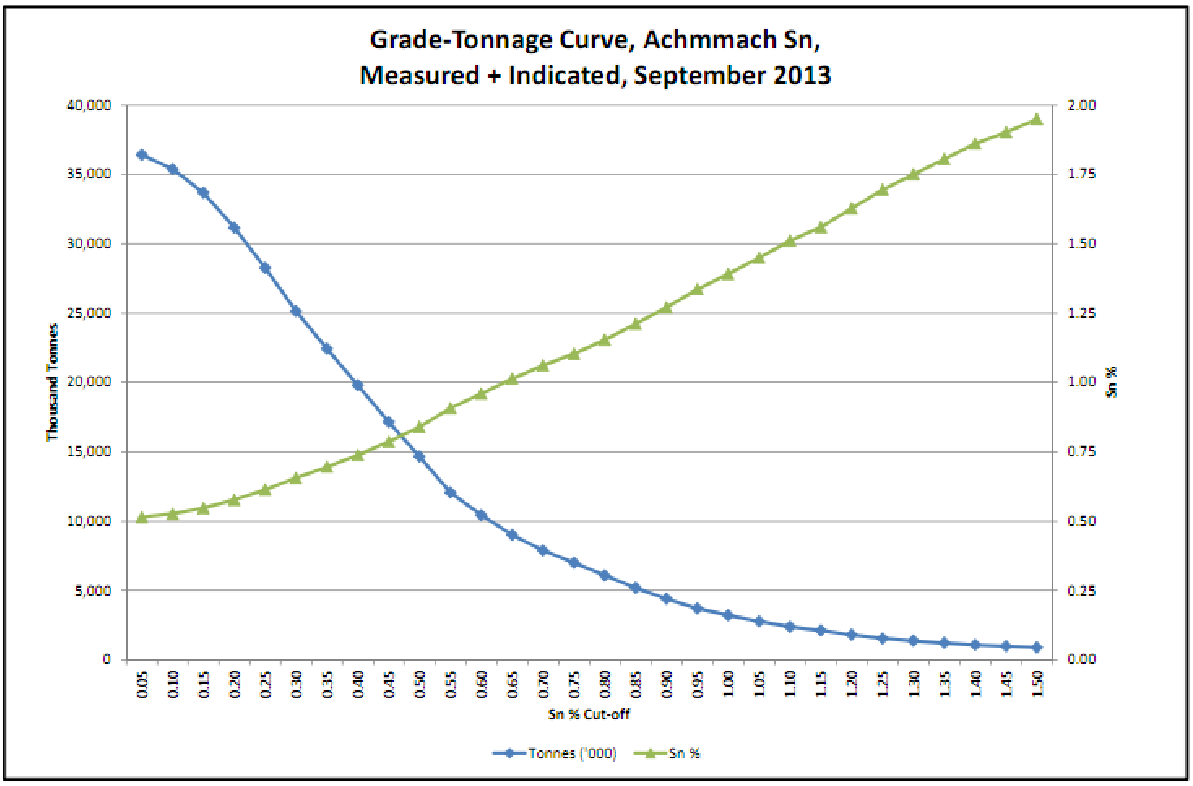


Figure 3-46 Grade tonnage curve for Meknès Trend

3.12.2 Western Zone (Sidi Addi Trend)

Table 3-32 shows total resource for the Western Zone, the grade tonnage curve is shown in Figure 3-47 below.

Table 3-32 Total Western Zone Resource

Classification	kTonnes	Sn%	Sn (kt)
Measured	-	-	-
Indicated	340	1.25	4.2
Inferred	-	-	-
Total	**340**	**1.25**	**4.2**

The Sn grade in this table has been rounded to the nearest 0.05% Sn.

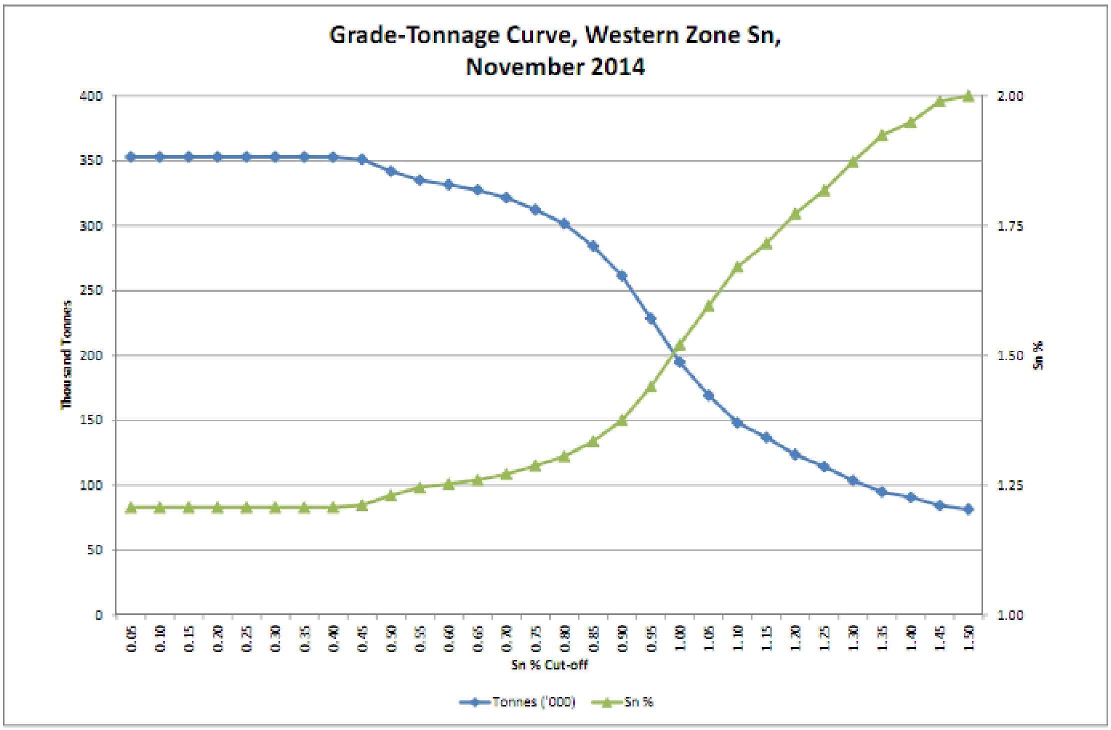


Figure 3-47 Grade tonnage curve for Western Zone

3.12.3 Total Combined Achmmach Tin Resource

The total combined Mineral Resource for the Achmmach Tin Project is presented in Table 3-33.

Table 3-33 Total Combined Mineral Resource – Achmmach Tin Project

Classification	Mtonnes	% Sn	kt Sn
Measured	1.6	1.0	16.1
Indicated	13.3	0.8	111.2
Inferred	-	-	-
Total	**14.9**	**0.85**	**127.3**

3.13 Comparison to Previous Estimates

3.13.1 Meknès Trend

Comparison to previous resource estimates is tabulated below (Table 3-34). The earliest model in December 2008 used inverse distance estimation methods and all subsequent modelling has used ordinary kriging. Cut-off for the earlier resource was quoted about 0.6% Sn whereas all other models used a 0.5% Sn cut-off for reporting of mineral resource estimates. Drilling in 2012 led to an increase in the global resource for the estimate in that year. In 2013, a further 103 holes were drilled but the focus was to increase confidence rather than increase the extents. Figure 3-48 shows the successful conversion to higher confidence resources.

Table 3-34 Comparison to previous estimates

Year	Estimator	Category	Mtonnes	% Sn	kt Sn
2008	Carras	Inferred	6.0	0.90	54.0
		Total	**6.0**	**0.90**	**54.0**
2010	QG	Indicated	2.2	0.80	17.6
		Inferred	4.8	0.80	38.4
		Total	**7.0**	**0.80**	**56.0**
2012	Mining One	Indicated	5.3	0.80	42.4
		Inferred	9.3	1.00	93.0
		Total	**14.6**	**0.90**	**131.4**
March 2013	QG	Measured	0.5	1.20	6.0
		Indicated	14.2	0.85	120.7
		Inferred	0.6	0.70	4.2
		Total	**15.3**	**0.85**	**130.1**
September 2013	QG	Measured	1.6	1.00	16.1
		Indicated	13.0	0.80	107.0
		Total	**14.6**	**0.85**	**123.1**

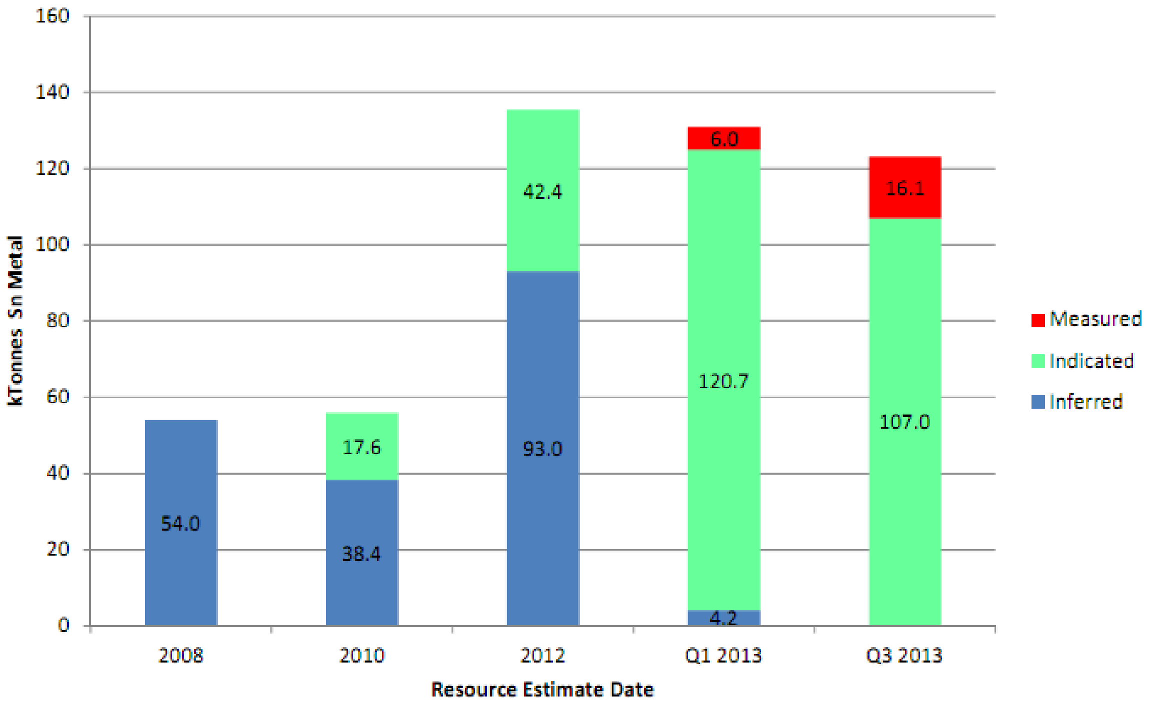


Figure 3-48 Resource comparison for contained Sn metal and category

3.13.2 Western Zone (Sidi Addi Trend)

Comparison to previous resource estimates is tabulated below. Tonnes, grade and contained metal have all increased due to further drilling at depth and the eastern extents (Table 3-35).

Table 3-35 Comparison to previous estimates

Indicated Resource	ktonnes	% Sn	kt Sn
Feb-14	221	0.95	2.1
Nov-14	340	1.25	4.2
Difference	119	0.30	2.1

3.14 References

Job, M. 2013. Quantitative Group Mineral Resource Report, Sept 2013.

Job, M. 2014. Achmmach Western Zone Resource Estimate, Morocco.

4 GEOTECHNICAL

4.1 Hydrogeology

A hydrogeological research programme was initiated in 2010 by engaging Golder to carry out a 2D resistivity survey over selected sites in the project area. The survey concentrated on three locations where geological structures, incorporating dyke features appeared to be promising aquifers. As a result of the survey programme Golder was able to recommend a total of ten drill targets spread among three locations. Two of these locations lie within the Achmmach tenement PE2912 within 1 km of the Achmmach tin deposit.

Bureau de Recherches et de Participantions Minières BRPM (French: Bureau of Mining Research and Investments; Morocco) experienced water inflow into one of the underground exploration workings at an elevation of 890 mRL which ultimately filled the shaft to 945 mRL. This shaft was subsequently used as a water supply for exploration drilling, but by late 2009 ground water had stopped following into the shaft. Although representative of only a small section of the deposit, this experience suggests there may be perched aquifers which readily drain but are of relatively small volume. This view is supported by the lack of water encountered during drilling.

During late 2011, after consultation with representatives of the regional water agency Agence des Bassines Hydrauliques Sebou (ABHS), Atlas Tin engaged SOLROC, a local water boring company to drill a series of five bores at sites NE of Achmmach as recommended by Golder.

4.2 Seismicity and Stress Field

4.2.1 Seismicity

The Achmmach deposit is in a region of low earthquake hazard which is defined as having an acceleration coefficient of 0.8 m/s^2 ((USGS) see Figure 4-1). The Global Seismic Hazard Assessment Program (GSHAP) produces probabilistic maps which indicate the maximum peak ground acceleration for which there is a 10% chance of occurrence in 50 years. The peak ground acceleration of 0.8 m/s^2 or 0.08g indicates an earthquake for which there is noticeable shaking but likely to be little or no damage.

The largest earthquake recorded to date within 250 km of the proposed mining area was recorded on 24th February 2002, measuring 6.4 on the Richter scale and located at a depth of 12.6 km. The earthquakes that have been recorded in the area are located deep (>5 km) and not expected to have any impact on the underground mine or infrastructure.

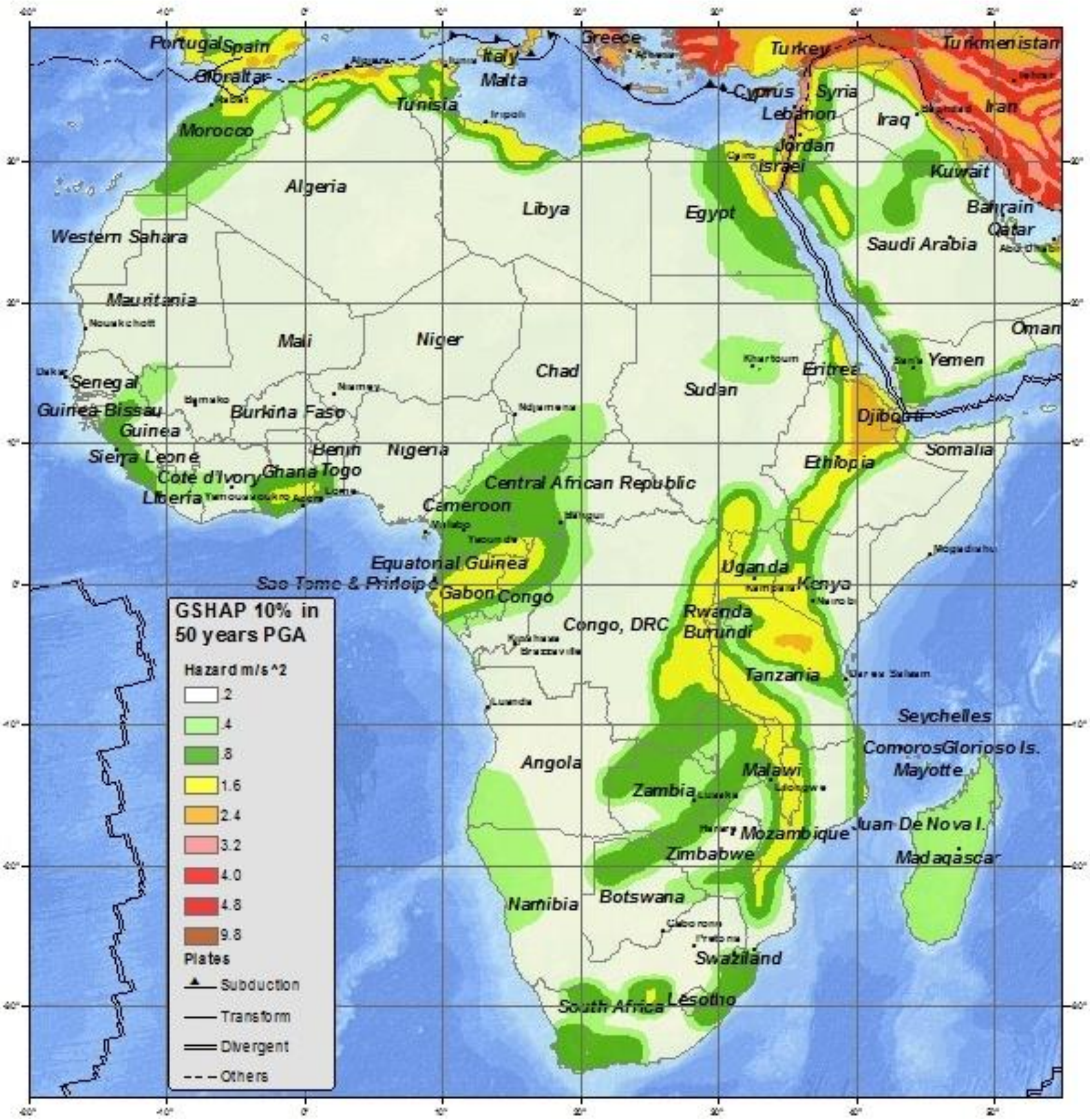


Figure 4-1 Seismic hazard map

4.2.2 Stress Field

The major stress orientations documented for Africa are presented in Figure 4-2. Various stress orientations have been measured along the northern boundaries of the African plate (GFZ German Research for Geoscience, 2009). The site structural data indicates a major principal stress field of approximately NW-SE in the Achmmach area and a strike-slip regime of $\sigma H > \sigma V > \sigma h$.

No stress measurements have been taken on site.

The major principal stress $\sigma 1$ ($\sigma 1$ corresponds to σH) is estimated to be orientated NW – SE, perpendicular to the major veins shears and near perpendicular to the shears and bedding / foliation. The minor principal stress ($\sigma 3$ or σh), is orientated sub-parallel to the veins, shears and bedding in an NE-SW direction, with the intermediate principal stress ($\sigma 2$ or σV), near vertical. Table 4-1 summarises the estimated stress regime.

Table 4-1 Estimated stress fields for Meknès and East.

Stress Magnitude	Trend	Plunge
σ 1 = σ 2 * 1.1	150°	15°
σ 2 = 0.027 * d	0°	75°
σ 3 = σ 2 * 0.9	60°	15°

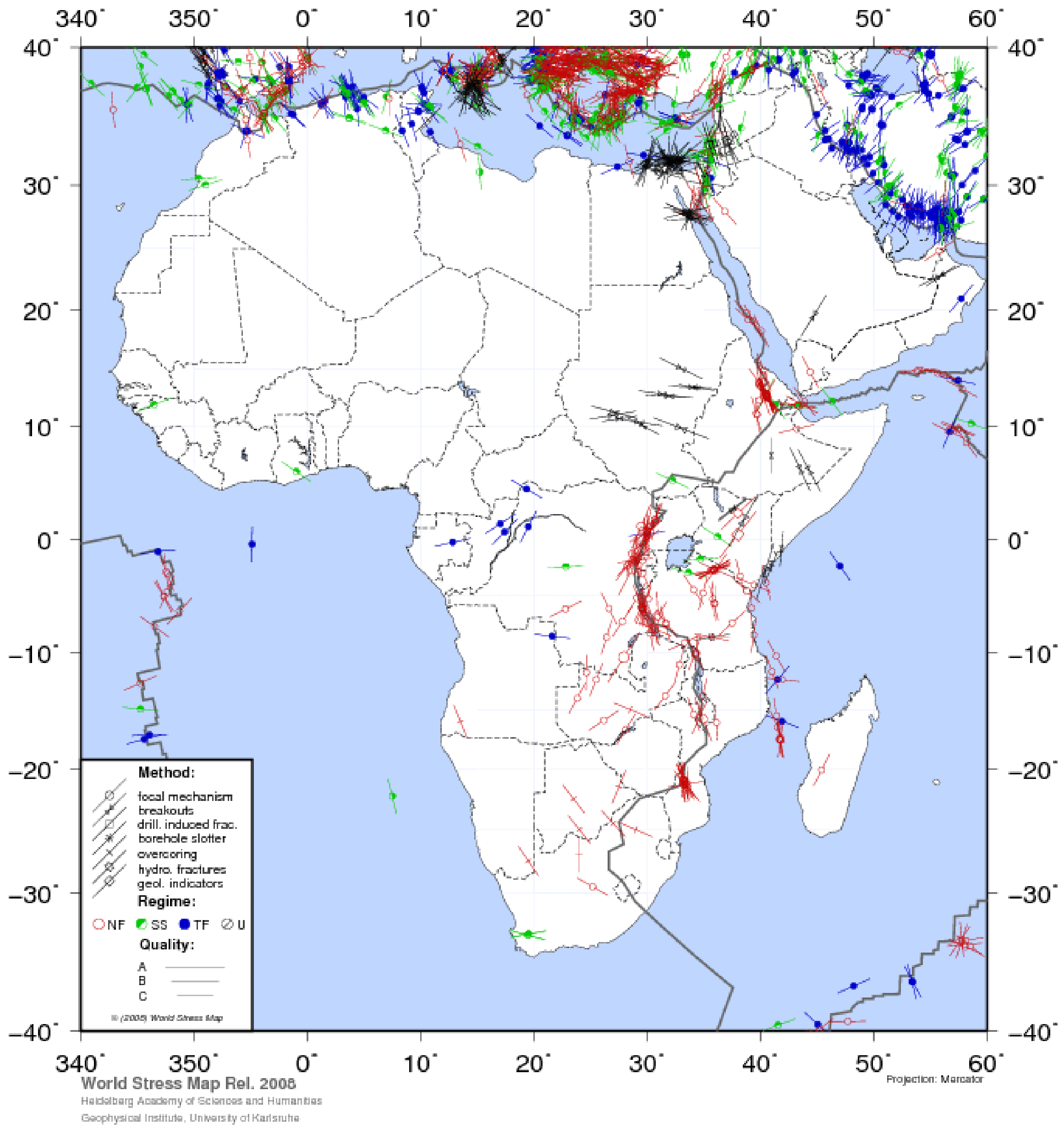


Figure 4-2 Major stress orientations in Africa.

4.3 Geotechnical Data

The geotechnical analysis was completed by Mining One Consultants (Appendix 4A) and is based on following geotechnical data:

- drill hole logs for AD001 – AD031, AD034 – AD130, S01 – S29 and SF1 – SF3
- geotechnical structural logs for AD119 – AD233
- drill core photography for holes (AD009-AD010, AD014-AD019, AD021-AD029, AD031, AD034-AD048, AD050-AD051, AD054, AD056, AD063, AD071, AD076, AD087, AD089, AD091, AD095D1, AD096, AD098, and AD100-AD233)
- geotechnical data for the portal and boxcut was provided as a summary report which included diamond drill logs, structural logs and core photos for holes GTD 01 – GTD 05. Note as built survey co-ordinates were not included in this report.

Geotechnical drill hole data was used to assess the rock mass quality and for the structural assessment. The project geologists recorded alpha and beta angles, however, most of the drill holes did not include all the geotechnical parameters required to complete a rock mass quality assessment.

It was also observed that logging intervals varied for each geological and geotechnical parameter which has resulted in additional handling of data to prepare for analyses. Logging intervals ranged from 0.1 m to 4.4 m, with 95% of the data defined in 3.1 m intervals or less and were based on drill run length and not on geotechnical similarity.

The key geotechnical parameters missing from the dataset included:

- number of joint sets (Jn)
- joint roughness (Jr)
- joint Infill (Ja)
- rock strength.

In order to compile a geotechnical dataset for rockmass classification:

- The number of joint sets (Jn) were interpreted from core photos and dip plots and then assigned to logged intervals.
- Joint alteration (Ja) values were assigned for each logged interval based on structural logging completed for that same interval.
- Joint roughness (Jr) values were assigned for each logged interval based on structural logging completed for that same interval (holes AD119 to AD130).
- The initial data set (holes AD119 to AD130) was used as a basis to expand the overall dataset by statistically determining the parameters not provided.
- Weathering data, where missing, was interpreted from core photography.
- Rock strength was estimated using a simple probabilistic approach considering rock type, weathering and alteration and is based on the initial data set (AD119 to AD130) where 64% of logged strength were of high to extremely high strength.
- Joint alteration (Ja) and Joint roughness (Jr) were estimated using a simple probabilistic approach for the remaining holes (AD009 to AD117).

The geotechnical assessment used a representative sample of holes along the strike of the mine that intersected a preliminary version of the mine design. The sections of drill core used for analysis included data 10 m into the hanging wall and footwall from the orebody boundary.

The following holes were used in the DFS to calculate the rock mass characteristics for:

- Meknès-AD132, AD133, AD134, AD135, AD145, AD155, AD156, AD159, AD161, AD166, AD207, AD216, AD221, AD222
- Eastern Zone-AD188, AD191, AD193, AD194, AD195, AD199, AD206, AD210, AD212, AD219, AD224.

Additional holes were drilled in the vicinity of the central and eastern box cuts (GTD01, 03, 04 and 05) to gain insight into the rock masses in which the box cuts, portals and declines will be developed. These holes were drilled, photographed, logged and sampled.

4.4 Rock Properties

A total of 68 core samples were collected for laboratory testing as part of the geotechnical study. Samples were collected under the guidance of the geotechnical engineer and then sent to the "Laboratorie de Structure & Rehabilitation (LSR)", a Moroccan rock mechanics laboratory. It was observed that due to the inter-bedded nature of the host rock, anisotropic properties were noted in most of the tests, resulting in a wide range of results.

4.4.1 Laboratory Testing

A total of 71 uniaxial compressive strength (UCS) rock samples and 17 triaxial compressive strength (TCS) were tested and the results are presented in Table 4-2.

The objective of the UCS test was to verify and calibrate field strength estimates, for the estimation of rock deformation characteristics and intact rock density. Triaxial testing allows for the determination of rock mass strength friction angle and cohesion from plotting Mohr Coulomb non-linear curves and tangent straight-line strength envelopes.

Typically, five to ten samples which are representative of each of the likely rock type domains across the strike of the mine design are required to obtain a reasonable population. Of the 88 tests completed, 59 of the UCS tests and 11 of the TCS tests were to a satisfactory standard.

Table 4-2 Laboratory Tests

Hole ID	Mining Area	Stratigraphy	Laboratory Tests (UCS)	Laboratory Tests (TCS)
AD107	Meknès	Sandstone	9	-
AD153	Meknès	Mudstone	2	-
AD174	Meknès	Sandstone	6	3
AD183	Eastern Zone	Sandstone	1	-
AD184	Eastern Zone	Cataclastic	-	1
AD187	Eastern Zone	Sandstone/ Cataclastic	5	1
AD191	Eastern Zone	Sandstone	1	-
AD193	Eastern Zone	Mudstone	2	-
AD202	Eastern Zone	Sandstone	4	1
AD205	Eastern Zone	Mudstone/ Sandstone	5	2
AD206	Eastern Zone	Shale	8	4
AD212	Eastern Zone	Shale	3	-

AD221	Meknès	Sandstone	1	1
AD225	Meknès	Sandstone	5	-
AD248	Meknès	Sandstone	17	4

4.4.2 Uniaxial Testing and Results

UCS testing comprised single-stage testing. The results display a wide range of strengths from weak to medium strong in mudstone, weak to very strong in sandstone, weak to very strong in shale and strong to very strong in siltstone. The results are presented in Table 4-3.

The dominant failure styles were shear failures through the intact rock and as such many of the UCS results are a measurement of the shear strength of the discontinuities rather than the intact shear strength. Although discontinuities generally control rock mass behaviour, the UCS results do not provide a true value of the intact rock strength. Table 4-3 below summarises the UCS tests for each rock type.

Table 4-3 UCS Results

Statistics	Mudstone	Sandstone	Shale	Siltstone
No. of tests	7	42	15	3
Minimum (MPa)	12.6	5	7	91
25% Quartile (MPa)	15	23	19	91
Weighted Mean (MPa)	26	43	72	212
Median (MPa)	25	38	72	245
75% Quartile (MPa)	33	60.5	91	274
Maximum (MPa)	45	161.4	274	274

The results from the test work are presented as a frequency distribution chart shown in Figure 4-3 for fresh rock samples. The results for the UCS tests on fresh rock indicate that approximately 70% of the samples returned a UCS less than 50 MPa and approximately 30% of the samples returned a UCS value greater than 60 MPa. The variance is high in all rock types, reflected by the 25% Quartile to 75% Quartile range. The shape of the frequency distribution suggests a bimodal distribution which may be indicative of a rock strength anisotropy, associated with a well-developed rock fabric (i.e. bedding and foliation); the lower strength mode reflecting rock strength in a direction which is parallel to bedding and the higher strength mode reflecting rock strength in a direction which is orthogonal to the bedding. An average UCS value of 75 MPa, a minimum of 50 MPa and a maximum of 100 MPa were adopted for design purposes. This was based on laboratory results and published typical physical properties for tested rock types.

The major mode has a mean of 23 MPa, whilst the minor mode has a mean value of 110 MPa.

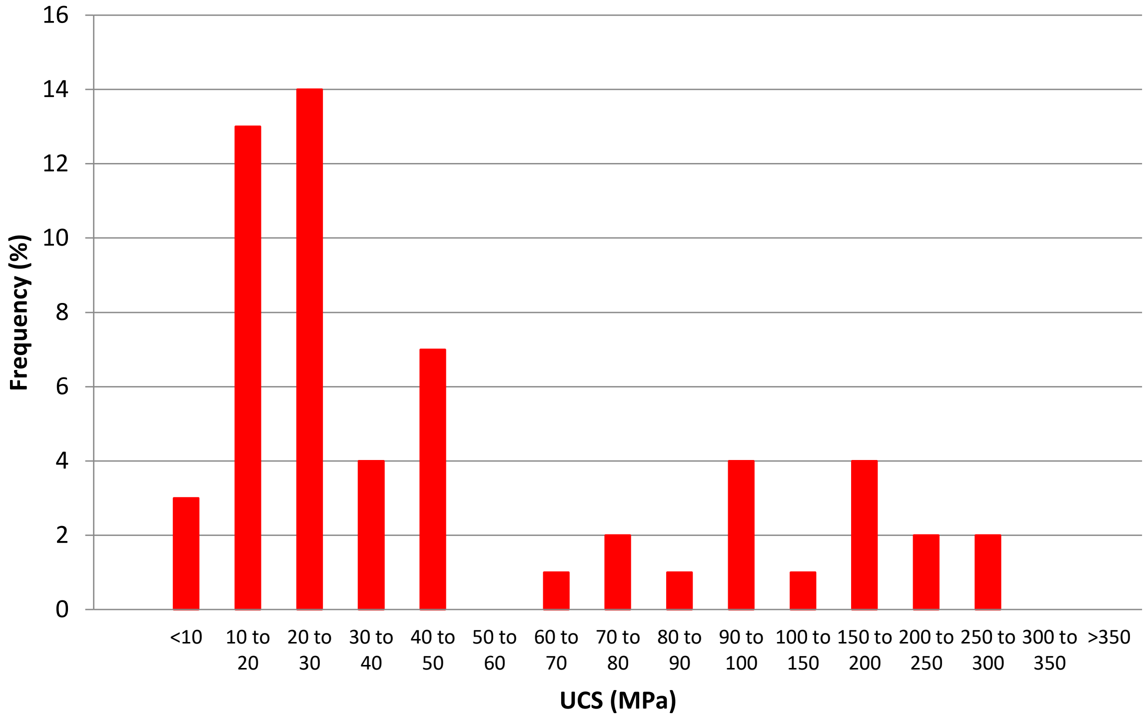


Figure 4-3 Uniaxial compressive strength (UCS); fresh rock, all lithologies

Analysis has shown that the foliation and bedding planes caused non-symmetric deformation and their behaviour is unpredictable. Goshtasbi, Ahmadi, & Seyedi (2006) showed that the minimum strength value of anisotropic rocks is at an orientation angle of β = ±30° (β is the angle between the axis of the core and foliation or bedding) and the maximum failure strength is either at β = ±0° or β = ±90°.

Based on these UCS data sets and anisotropy graphs it was estimated that the UCS values for the orientation (β) angle 20° to 40° is representative of the rock mass strength parallel to the foliation or bedding. The UCS values for the orientation (β) angle 0° to 20° and 40° to 90° are representative of the rock mass strength in failure directions non-parallel to the foliation or bedding (Goshtasbi, Ahmadi, & Seyedi, 2006). Table 4-4 summarises these results.

Table 4-4 Estimated UCS data for different orientation angles

Orientation Angle (β)	Lower Quartile (MPa)	Ave (MPa)	Upper Quartile (MPa)
20° to 40°	50	75	100
0° to 20°; 40° to 90°	88	134	161

Anisotropy is expected to significantly influence stope and pillar stability. The maximum estimated UCS is perpendicular to the bedding/foliation and ranges between 88 – 161 MPa (Mean 134 MPa). This perpendicular strength would be applicable to pillar strength designs.

The bedding / foliation parallel strengths, ranging between 50 – 100 MPa (Mean 75 MPa), would be applicable to hangingwall and footwall strength designs where the stresses are parallel to the stope. The parallel strengths are also an indication of the strong bonds between the bedding and foliation planes.

A value of UCS value of 134MPa will be used in this report for rock mass characteristics and pillar strength calculations.

4.4.3 Rock Mass Shear Strength

An assessment of the Hoek Brown rock mass strength was carried out using Rocscience RocLab software. The development and basis of this program is described elsewhere (see Barton & Brandis (1980), Hoek, Carranza-Torres, & Corkum (2002), Hoek (2013) and Hoek, Hoeks Corner). Using the intact rock strength data from UCS and the rock mass data (GSI) from core logging and a range of other factors a rock mass strength can be estimated. The key parameters and results are summarised in Table 4-5. The Hoek Brown strength uses the following parameters:

- Intact Rock for the orebody UCS is 134 MPa and the waste rock UCS is 75 MPa (inferred from laboratory UCS test results).
- GSI of 58 for the waste rock and 59 for the orebody, the mean for the rock mass (determined by RMR calculations from core logging).
- Intact Rock Factor mi of 15 for the waste rock and mi of 20 for the orebody (determined from industry data for sandstone and quartz veins).
- Disturbance Factor D of 0 is appropriate for undisturbed rock mass conditions deeper from the stope.

The Hoek-Brown strengths are defined by curved failure envelopes over which straight-line (Mohr-Coulomb) strengths are fitted. Rock mass strengths are estimated to be, for mining depths of 50 m to 500 m below surface.

Table 4-5 RocLab Data for a disturbed Rock Mass (D = 0).

		Waste Rock	Orebody
UCS (MPa)		75	134
GSI		58	59
mi		15	20
D		0	0
Mohr-Coulomb	Cohesion (kPa)	1,995	2,801
	Friction (°)	48	55
Hoek-Brown	mb	3.347	4.625
	s	0.0094	0.0105
	a	0.503	0.503

4.4.4 Triaxial Compressive Strength (TCS) Testing and Results

Triaxial tests were undertaken in the laboratory to assess the intact shear strength. Hoek triaxial multistage testing was undertaken on 17 fresh rock samples from all lithologies.

The rock mass is currently classified geologically as a suite of inter-bedded sandstones, shales and mudstones and to date it has not been possible to delineate the regions comprised of each lithology. As such all TCS data has been combined and interpreted based on weathering class only.

A linear best-fit has been interpreted and passes through the data points. The results along with C' and ϕ values obtained from the Rocscience program Roclab are presented below in Table 4-6.

Table 4-6 Triaxial Results

	UCS (MPa)	D	GSI	mi	Unit Weight (MN/m3)	C (kPa)	Friction (degrees)
All Lithologies RocLab	8.0	0.0	57	50	0.027	943	43.0
All Lithologies RocLab	8.0	0.5	57	50	0.027	796	38.7
P-Q Plot All Lithologies						59	32.0
Sandstone RocLab	10.4	0.0	57	50	0.027	1,030	45.0
Sandstone RocLab	10.4	0.5	57	50	0.027	871	40.8
Industry Sandstone RocLab	75.0	0.0	57	18	0.027	1,674	51.5
Industry Sandstone RocLab	75.0	0.5	57	18	0.027	1,355	47.7
Sandstone P-Q Plot						1,000	58.9
Shale RocLab	6.7	0.0	58	50	0.028	909	41.3
Shale RocLab	6.7	0.5	58	50	0.028	766	37.0
Industry Shale RocLab	35.0	0.0	58	6	0.027	1,013	36.7
Industry Shale RocLab	35.0	0.5	58	6	0.027	790	32.7
Shale P-Q Plot						587	60.5

4.4.5 Rocscience RocLab Evaluation

Triaxial laboratory test data were analysed with the Rocscience Roclab program. The interpretations are presented in Figure 4-4 and Figure 4-5; and Table 4-7 and Table 4-8 for the hard rock and weathered rock.

Hard Rock RocLab Parameters

Table 4-7 Rock strength parameters for hard rock

UCS (MPa)	D	GSI	mi	C (kPa)	Friction (°)
75	0.4	55	7	1183	40

Young’s modulus (GPa)	υ	mb	s	a	Tensile
15-25	0.25 – 0.35	0.939	0.0031	0.504	-0.3MPa

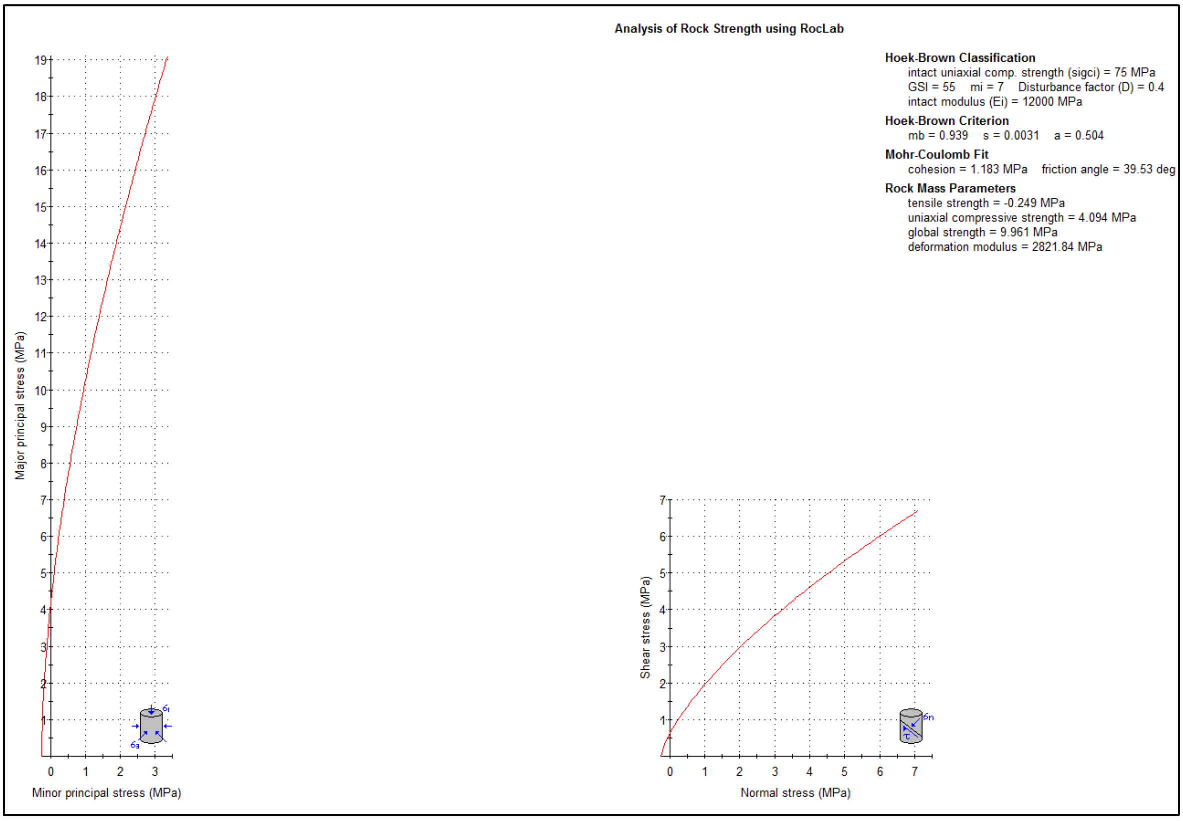


Figure 4-4 Rock Strength Analysis for the Hard Rock

Weathered Rock RocLab Parameters

Table 4-8 Rock strength parameters for weathered rock

UCS (MPa)	D	GSI	mi	C (kPa)	Friction (°)
27	0	30	117	115	34

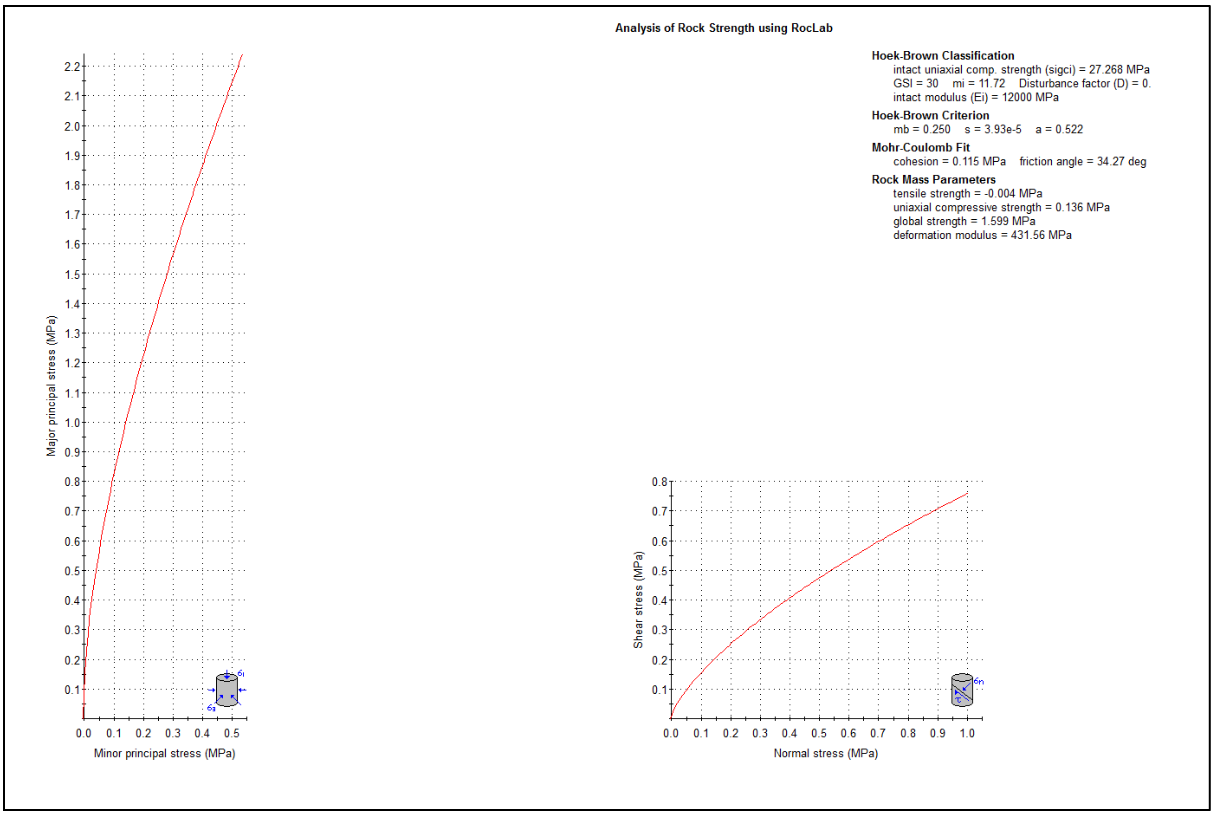


Figure 4-5 Rock Strength Analysis for the Weathered Rock

4.5 Structural Data

In fresh rock, discontinuities play a significant role in the stability of excavations. Discontinuities weaken a rock mass making it more susceptible to failure and create planes of weakness on which structural instability may occur. It is therefore important to identify all possible discontinuity sets.

An analysis of the structural data from the orientated drill core included:

- bedding and foliation
- joints
- shears
- veins

Resource holes have been drilled in north-south direction, perpendicular the ore body. Whilst optimum for grade estimation, this predominant orientation tends to create a bias in the recorded structural data because structures with a dip and dip direction parallel to the axis of the drill hole tend to not to be intercepted. This is most apparent in the logged joint structures and has resulted in a data shadow on the stereographic projection.

Furthermore, the orientation data of the logged joints and shears have a very similar orientation to that of the bedding, which has reduced the visual significance of the lower occurring joint and shear sets.

This combination of factors has inhibited the identification of joint sets using solely logged data and therefore mapping data has been relied upon to account for joint sets with a lower occurrence.

Comparisons were made between geotechnical logged boreholes and mapped underground excavations and interpretations from previous geological studies and presented in Figure 4-6 to Figure 4-9.

For the underground wedge analysis, the joint and shear characteristics were adopted from interpretation of the mapping data and the bedding orientation was adopted from the bore hole logged data.

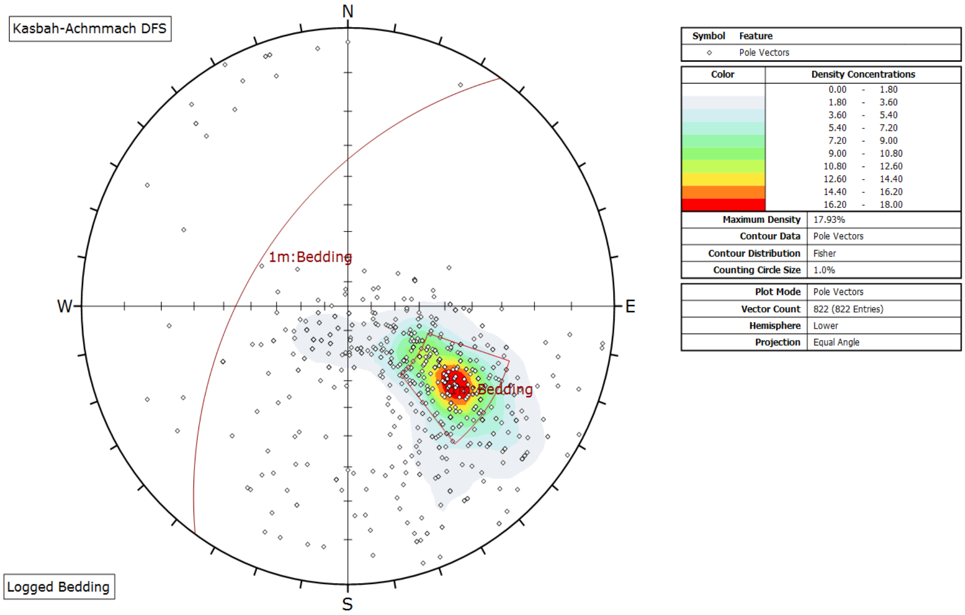


Figure 4-6 Bedding and foliation logged data

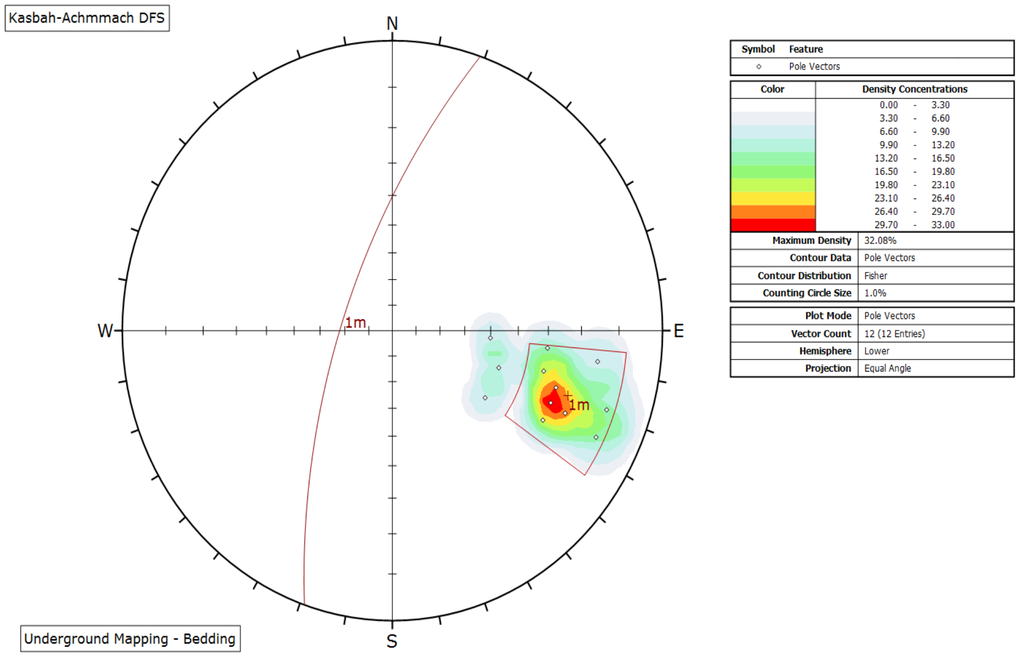


Figure 4-7 Bedding and foliation mapped data

Both geotechnical mapping and logging indicates a strong bedding/foliation set. This structural feature is the most prominent and will play a significant role in excavation stability.

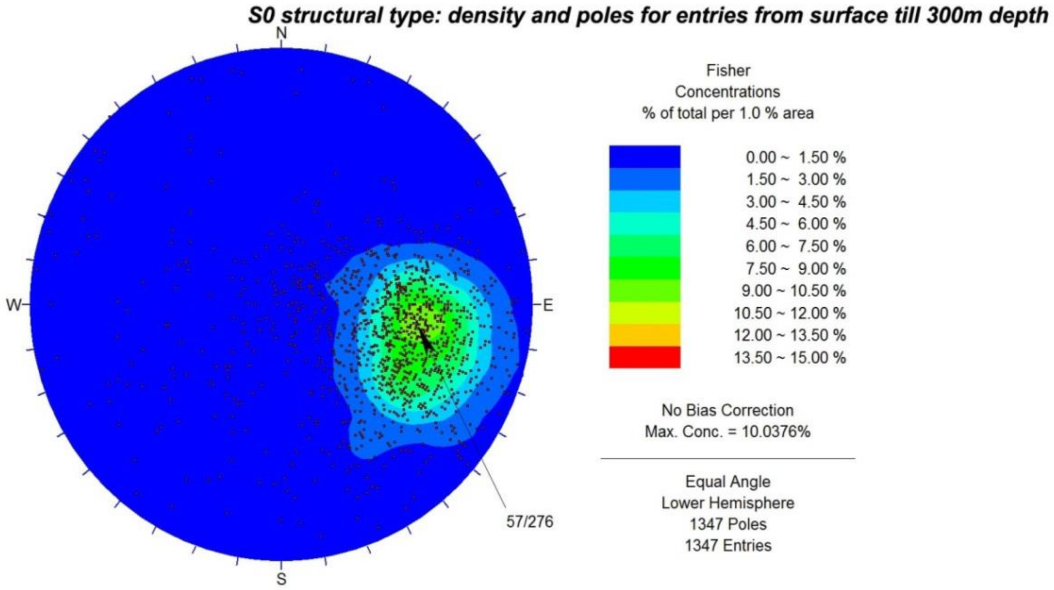


Figure 4-8 Distribution of bedding and foliation surface to 300m depth

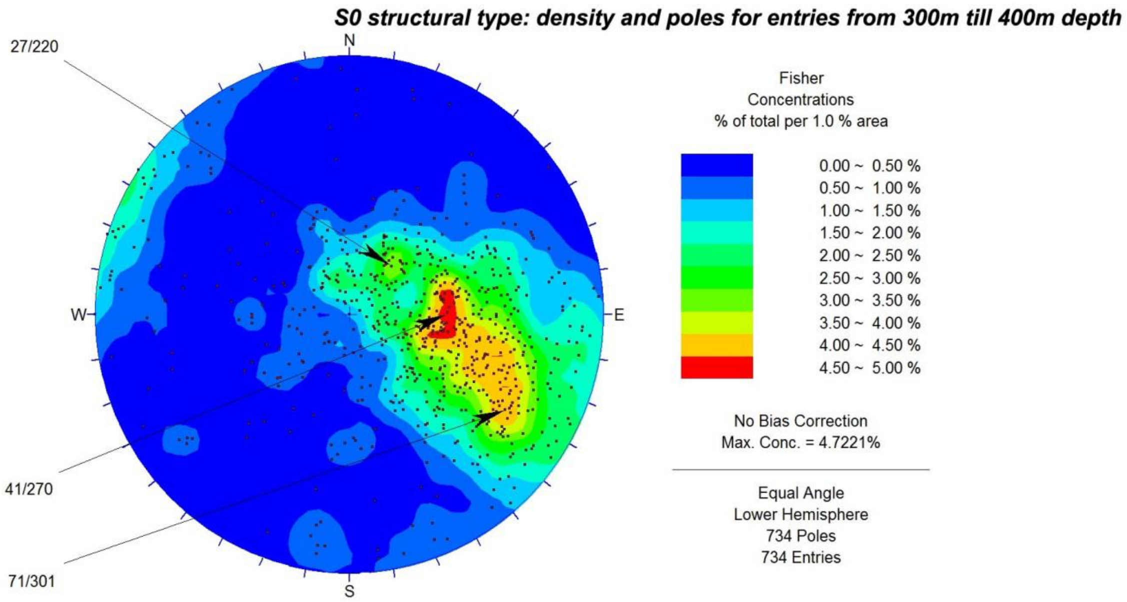


Figure 4-9 Distribution of bedding and foliation 300-400m depth

Changes in distribution of bedding and foliation have been interpreted over depth. A greater variance in dip and strike have been observed for depths 300 to 400 m than for depths <300 m below surface.

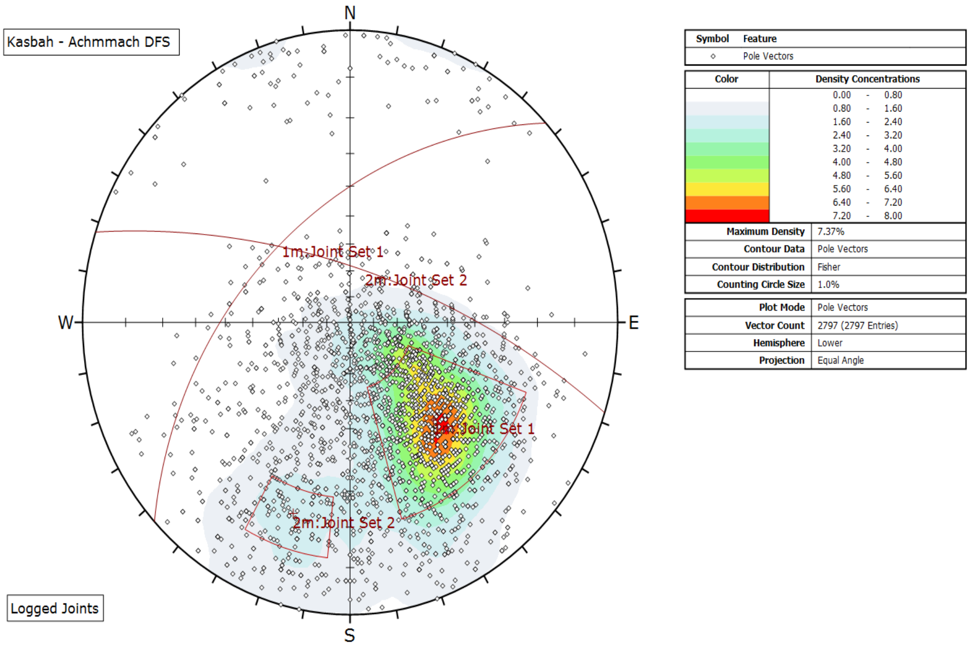


Figure 4-10 Joint sets logged data

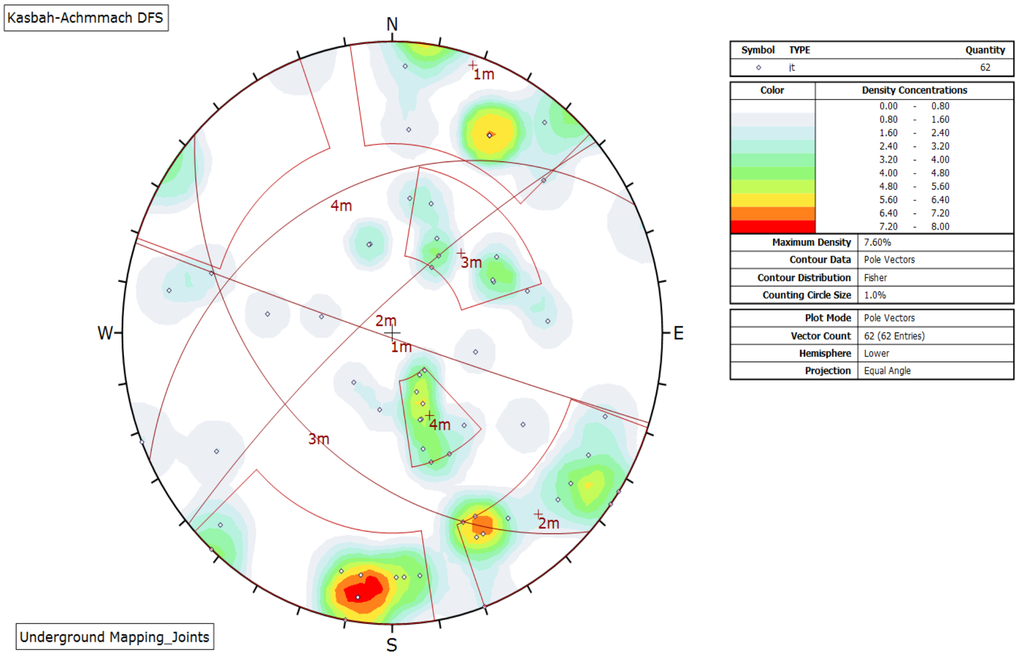


Figure 4-11 Joint sets mapped data

Joint sets logged in geotechnical boreholes were not as clearly identifiable as those logged in underground mapping. Due to the dominant presence of the bedding and foliation, joint data seemed to be incorporated into this data set and vice versa. The shadow effect, due to the unilateral direction of drilling can clearly be seen in Figure 4-10 and Figure 4-11. The underground mapping data set was used for the structural analysis.

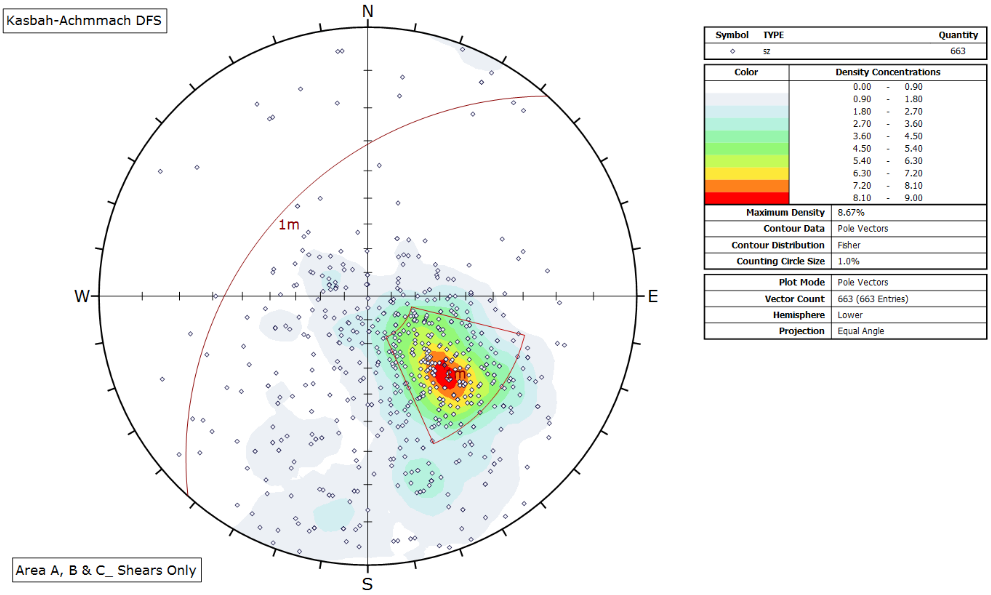


Figure 4-12 Shears logged data

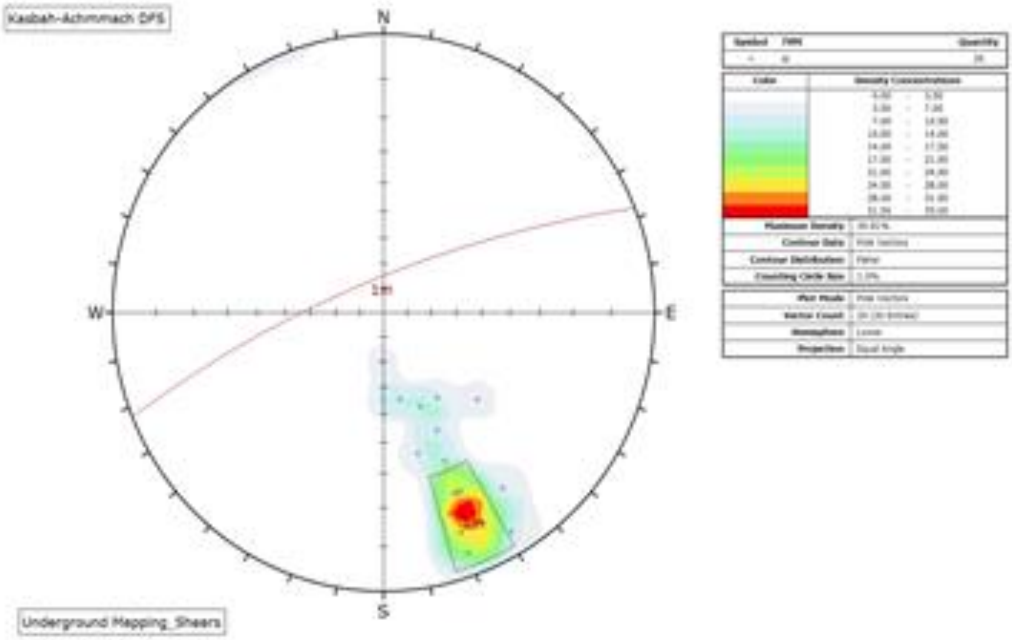


Figure 4-13 Shears mapped data

Shears for both the geotechnical logging and underground mapping were very similar to the bedding and foliation data set (Figure 4-12 and Figure 4-13).

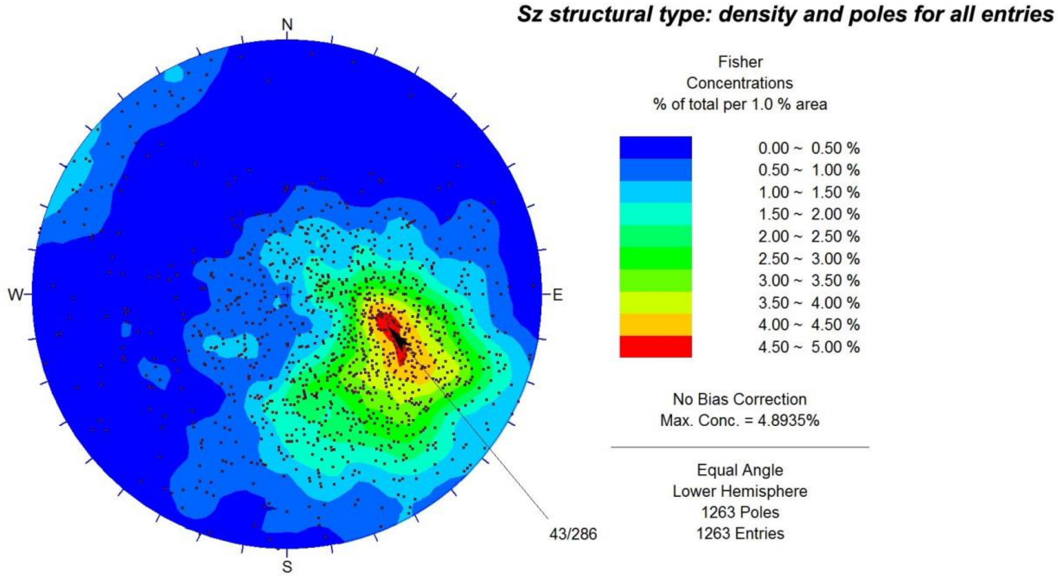


Figure 4-14 Structural interpretation of shears (Lindhorst)

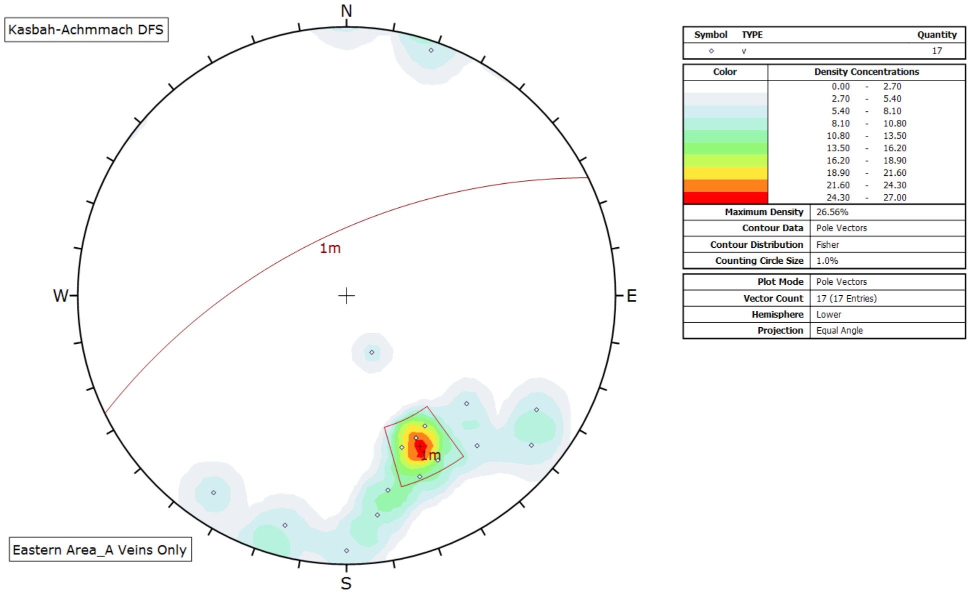


Figure 4-15 Veins logged

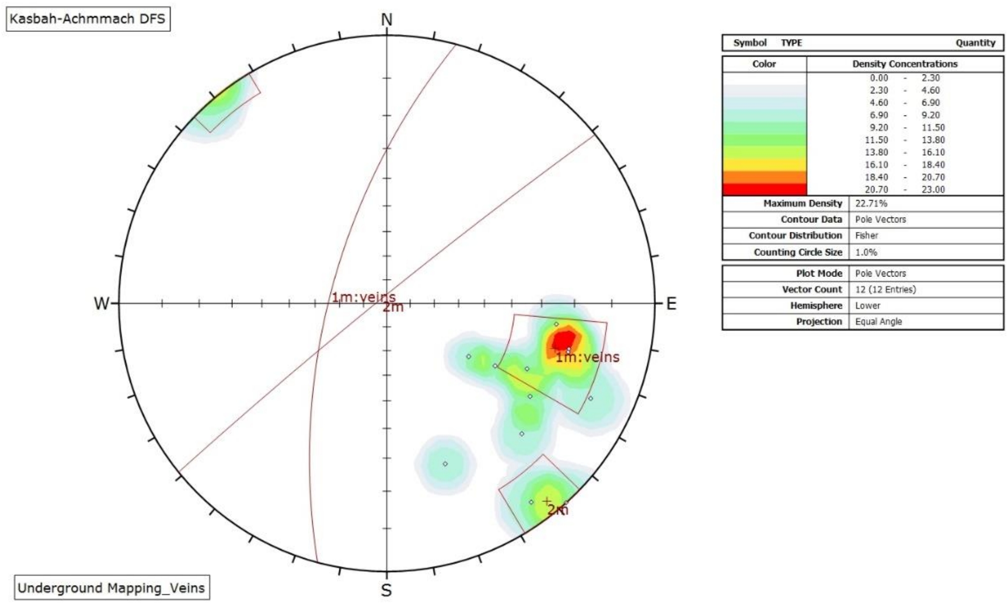


Figure 4-16 Veins mapped

Veins from both the geotechnical logging and underground mapping indicates similar trends.

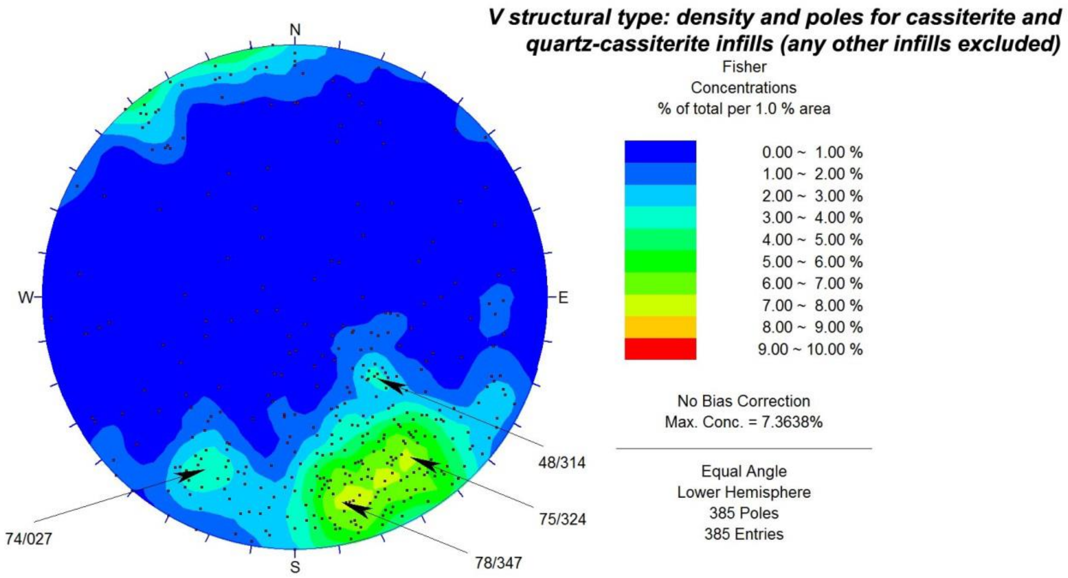


Figure 4-17 Structural interpretation of veins (Lindhorst)

An analysis of the structural data from the orientated drill core and from historical geological mapping is summarised in Table 4-9.

Table 4-9 Regional joint orientations.

Features	Strike	Dip Dir	Dip	Type
Bedding	NNE-SSW (20°)	WNW 275°-305°	40° – 80°	Bedding / Foliation
Joint 1	ESE – WNW (100°)	S 170°-225° N 350°-45°	70° – 90° 70° – 90°	Conjugate
Joint 2	NE – SW (30°)	SE 110°-160° NW 290°-340°	70° – 90° 70° – 90°	Conjugate
Joint 3	NW-SE (120º)	SW (190 º-250º)	30°-60°	Intermediate Dip
Joint 4	NE-SW (60º)	NW (315 º-350º)	20°-50°	Intermediate Dip
Shears	NNE-SSW (10º)	W (275 º-300º)	50°-80°	Steep
Veins	NE-SW (60º)	NW (330 º-345º)	65°-85°	Steep

Note that the bedding is the most prominent feature in the Achmmach project area and will therefore dominate the structural stability of any excavation.

4.6 Rock Mass Classification

To evaluate the rock mass, three main rock engineering classification systems were used. The three systems (Q, RMR and MRMR) are described below.

4.6.1 Rock Tunnelling Quality Index (Q)

The Rock Tunnelling Quality Index Q' is calculated using the following expression:

$$Q = \frac{\mathrm{RQD}}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{\mathrm{SRF}}$$

Where:

RQD is the rock quality designation for each interval

Jn is the number of defect sets

Jr is the joint (defect) roughness

Ja is the joint (defect) alteration number which is determined by the type and thickness of infill

Jw is the joint water reduction factor

SRF is the stress reduction factor.

The stress reduction factor (SRF) and joint water reduction factor (Jw), are required to be evaluated. It is assumed that dry conditions (<5 litres/min) occur for each of the domains, allowing Jw to be assigned a value of one (1). The SRF was estimated using Bartons Q Tables, rather than using a stress-strength relationship. An SRF value of 2.5 was used which is considered appropriate for a low stress environment.

The Modified Rock Tunnelling Quality Index Q' is modified from the Rock Tunnelling Quality Index Q by dropping the active stress quotient. Therefore, the Modified Rock Tunnelling Quality Index Q' is calculated using the following expression:

$$Q' = \frac{RQD}{Jn} \times \frac{Jr}{Ja}$$

4.6.2 Geomechanics Classification of Rock Mass Rating (RMR89)

Bieniawski's RMR'89 classification was also used to provide quantitative values representing the rock mass competency into rock engineering design methods.

The parameters included in the system are:

- rock material strength (UCS)
- RQD
- joint spacing
- joint roughness and separation
- groundwater

The GSI (Geological Strength Index) is a parameter used in the calculations of rock mass strengths and is expressed by the equation:

GSI = RMR – 5

The Q and RMR systems could be compared using Table 4-10.

Table 4-10 Ratings for rock mass classification systems

	Very Poor	Poor	Fair	Good	Very Good
Q	0.1-1	1-4	4-10	10-40	40 – 100
RMR	0-20	21-40	41-60	61-80	81-100

4.6.3 Mining Rock Mass Rating (MRMR)

The mining rock mass rating (MRMR) was developed by Laubscher and specifically used for the mining environment. This system takes into account the same parameters as the Geomechanics system, but combines the groundwater and joint condition, resulting in the four parameters:

- intact rock strength
- RQD
- joint spacing
- joint condition and ground water

Adjustments are applied to the MRMR value to take account of weathering of the rock mass, joint orientation relative to the excavation, mining-induced stresses and blasting effects. These values are used to determine stable spans in the open stopes.

4.7 Geotechnical Domains

The various rock mass rating methods were analysed using the 3D modelling program Leapfrog, to determine the geotechnical domains at Achmmach.

Based on these assessments the Achmmach project area was sub-divided into six geotechnical domains which included:

- Meknès Footwall Zone
- Meknès Ore Zone
- Meknès Hangingwall Zone
- Eastern Zone Footwall Zone
- Eastern Zone Ore Zone
- Eastern Zone Hangingwall Zone.

Table 4-11 is a compilation of all the rock mass ratings for the identified geotechnical domains.

Table 4-11 Rock mass ratings by method per domain

Domain	Statistics	Q	Q'	RMR	GSI	MRMR
Meknès HW	25%	1.1	2.9	57	52	43
	Ave.	3.4	8.9	62	56	43
	75%	6.5	16.5	69	64	52
Meknès Ore	25%	1.3	3.6	57	53	43
	Ave.	2.9	7.8	63	57	47
	75%	5.5	14	67	62	50
Meknès FW	25%	2.2	6.2	59	55	44
	Ave.	4.7	11.8	65	59	49
	75%	9.2	22.7	69	64	52
Eastern HW	25%	2.8	7	60	55	45
	Ave.	4.6	11.4	63	57	47
	75%	10.3	25.6	65	60	49
Eastern Ore	25%	3.7	9.1	63	58	47
	Ave.	5.2	13.0	65	59	49
	75%	8.5	21.3	66	61	50
Eastern FW	25%	3.5	8.6	61	56	46
	Ave.	5.0	12.5	63	58	47
	75%	9.1	22.8	66	61	50

Table 4-12 Ratings for Rock Mass Classification Systems.

	Very Poor	Poor	Fair	Good	Very Good
Q	0.1-1	1-4	4-10	10-40	40 – 100
RMR	0-20	21-40	41-60	61-80	81-100

A re-evaluation of the core logging and quality of data collected indicated that the RMR evaluation process appeared to be more complete than the Q evaluation and more representative of a mining environment. It was therefore decided that the RMR values will be used to further evaluate the following design parameters:

GSI based on the formula GSI = RMR – 5 (Bieniawski, 1989)

Q based on the formula RMR = 9lnQ + 44 (Bieniawski, 1989); and

N' Stability Number ((Potvin, 1988) (Mathews, et al., 1980)).

Table 4-12 provides the observed or logged results, while Table 4-13 provides a calculated result of Q, Q' N' and GSI based on the more reliable RMR rating. The results presented in Table 4-13 were used to develop further the design guidelines.

Table 4-13 Rock mass rating methods per domain.

Domain	Statistics	Q	Q'	RMR	GSI	MRMR
Meknès HW	25%	4.2	10.6	57	52	43
	Ave.	7.4	18.5	62	56	43
	75%	16.1	40.2	69	64	52
Meknès Ore	25%	4.2	10.6	57	53	43
	Ave.	8.3	20.6	63	57	47
	75%	12.9	32.2	67	62	50
Meknès FW	25%	5.3	13.2	59	55	44
	Ave.	10.3	25.8	65	59	49
	75%	16.1	40.2	69	64	52
Eastern HW	25%	5.9	14.8	60	55	45
	Ave.	8.3	20.6	63	57	47
	75%	10.3	25.8	65	60	49
Eastern Ore	25%	8.3	20.6	63	58	47
	Ave.	10.3	25.8	65	59	49
	75%	11.5	28.8	66	61	50
Eastern FW	25%	6.6	16.5	61	56	46
	Ave.	8.3	20.6	63	58	47
	75%	11.5	28.8	66	61	50

4.8 Ground Conditions Expected in Mining

The proposed mine plan involves two long hole stoping methods:

- bottom-up method with cemented rock fill (CRF)
- top-down open stope method with rib and sill pillars.

Broadly speaking the CRF method is planned to be used in the Central Meknès Zone while open stoping methods will be used in the peripheral areas – predominantly what is termed the Eastern Zone.

The stoping analysis addressed the following key areas:

- analysis of the expected stress environment
- evaluation of the rock mass ratings in the Meknès and Eastern Zones
- evaluation of the stope design parameters e.g. Stability Number and Hydraulic Radius
- evaluation of potential pillar layouts
- recommendations for CRF specifications
- evaluation of the proposed mining methods
- estimation of the expected mining dilution
- development of a geotechnical stress model utilising the FLAC 3D computer program to evaluate overall mine stability.

The geotechnical model is based on an integrated method where geological, hydrogeology, rock mass and structural data was used to construct the geotechnical model.

Based on the available information (surface exposure, geotechnical boreholes, laboratory testing) the rock mass was divided and simplified to conform to a number of basic mining rock types (Table 4-14).

Table 4-14 Basic mining rock types at the Achmmach Project.

Rock Type	Description
Soil / Scree (S)	Soil / Scree (thickness 1 m – 4 m)
Saprock (SAP) Weathered Rock	Saprock at various levels of weathering. (Moderately-Slightly) (Thickness < 5 m)
Fresh Rock	Host Rock: Inter-bedded shale, siltstone and sandstone. Ore: Complex intercalated layers of black tourmaline and milky white quartz veins within a matrix of host rock.

4.9 Rock Mass Classification Systems and Domains

To evaluate the rock mass, four main rock engineering classification systems were used: Q, RMR, MRMR and GSI. The details on these systems are provided in Bieniawski (1989) and Laubscher (1990).

4.10 Design Parameters

4.10.1 Stope and Pillar Design

The mine plan is for the Meknès Zone to be mined bottom-up mining with a Cemented Rock Fill (CRF) and the Eastern Zone to be mined top-down mining method with no fill. The stope design strategy was therefore to:

- Design stable stopes based on the Hydraulic Radius (HR) and the Modified Stability Number (N’) for the Meknès and Eastern Zones.
- Design stable pillars based on the Tributary Area Theory (TAT) and the Hedley and Grant pillar strength method for the Eastern Zone.
- Assess the indicative cemented rock fill strength required and provide some guidance on the specifications.

Stopes

Rock mass characterisation and stable hydraulic radius (HR) results have been derived using the Stability Graph Method (SGM) derived from Q’. This approach is commonly used for the consideration of stope design and support requirements. Its derivation is discussed in detail in rock engineering textbooks (eg (Potvin, 1988) (Mathews, et al., 1980)).

The SGM factors are termed the A, B and C factors and adjust for rock stress, critical joint orientation and gravity, respectively and according to variations in the stope geometry and defect orientations, as follows:

$$N' = Q' \times A \times B \times C$$

Where:

Q’ is the modified Rock Mass Quality index;

A-rock stress factor and is a function of the ratio of intact rock strength to induced strength and accounts for stresses acting on exposed surface of open stopes;

B-joint orientation factor is determined by the major geological structure assessed to be principally responsible for potential instability of the surface, and

C-gravity adjustment factor relates to the predicted mode of failure and the dip of the surface being considered.

The values for factors to be used in stope assessment are summarised in Table 4-15.

Table 4-15 Estimated factors for stability graph method.

	A	B	C
Backs / End Walls	0.3 – 1.0	0.5 – 0.8	2-5
Hanging and Foot Walls	0.4-1.0	0.5	5

The stability Number (N’) can then be used to determine the stable HR based on what has been achieved on similar with similar ratings. This is known as the Matthews Stability Graph Method (MSG) (Mathews, et al., 1980).

Stope Design Process

The MSG method is based on case studies. Here stope stability was assessed relative to the HR of the stope and the rockmass N’ classification. The interpretation of this data falls into three categories: stable, transition and caving.

The MSG method was used to determine the stable HR values based on the 25 – 75 percentile ranges of the Stability Number N’ calculated for unsupported rockmass for both the East and Central Meknès zones. The mean of the Stability Number N’ were used for calculating the stable stope sizes for unsupported and supported stopes. The results are shown graphically in Figure 4-18 and Figure 4-19.

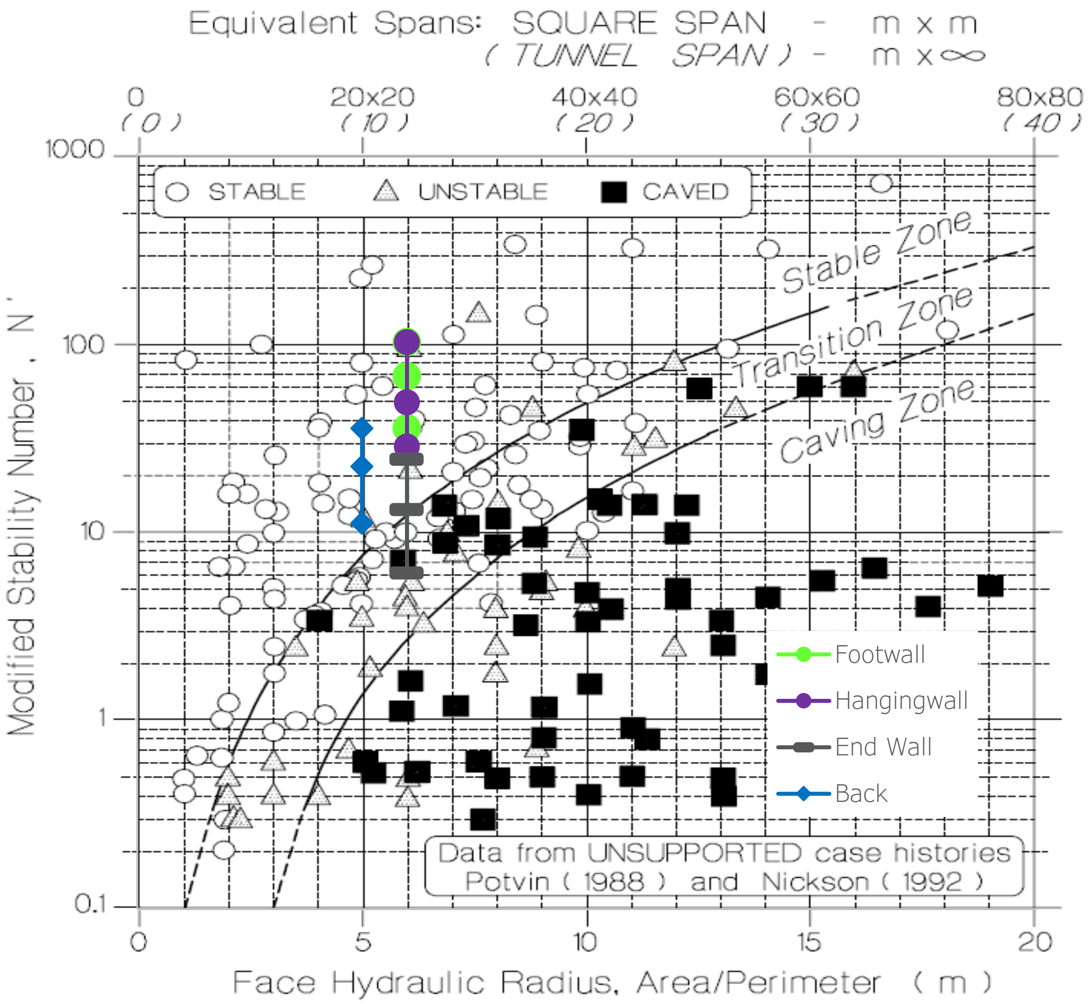


Figure 4-18 N' vs HR values for the unsupported stopes Meknès Zone (Hutchinson & Diederichs 1996).

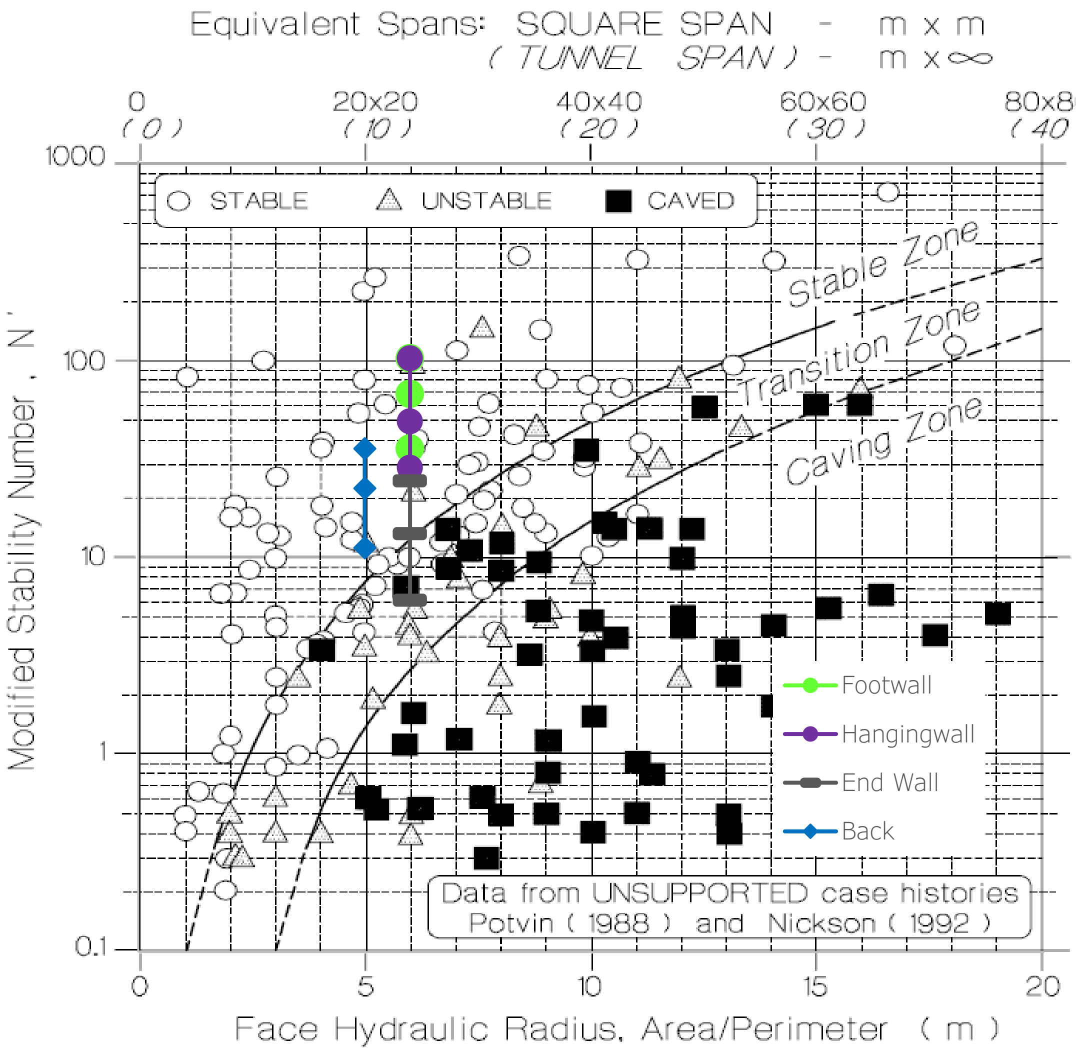


Figure 4-19 N' vs HR values for the unsupported stopes East Zone (Hutchinson & Diederichs 1996)

These results were then used to develop design guidelines. Plotting the stable stope strike lengths for the various back widths and wall heights allows the mine planner to determine optimum stope shape for maximising resource recovery. These design guidelines are presented as graphs, Figure 4-20 to Figure 4-23.

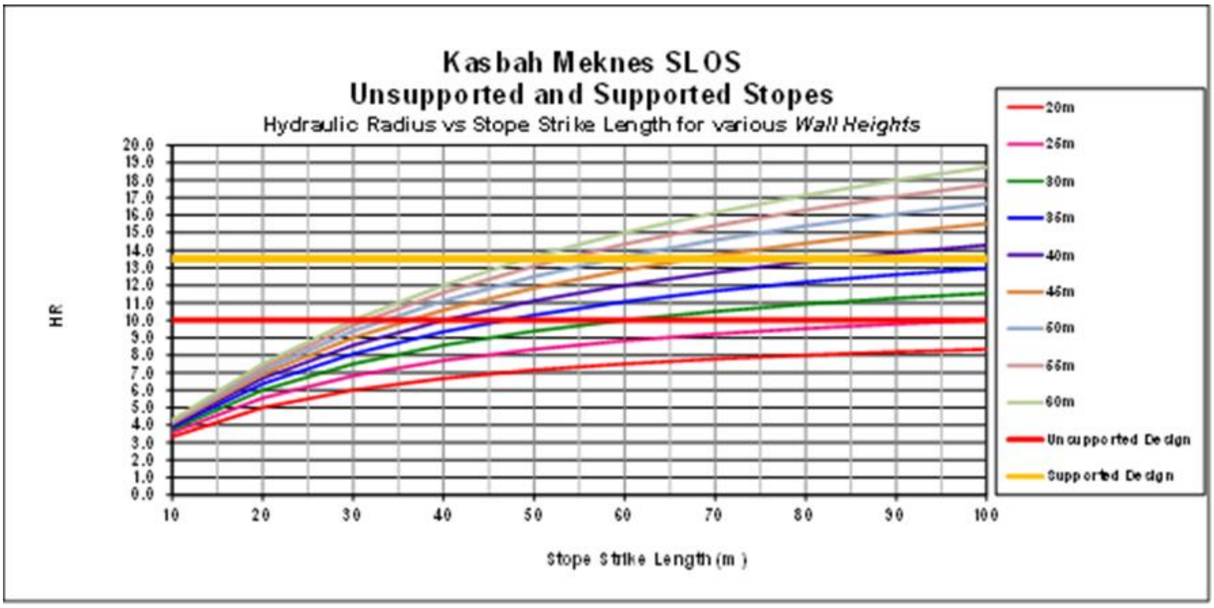


Figure 4-20 Meknès Zone-Design guidelines for stable unsupported and supported backs.

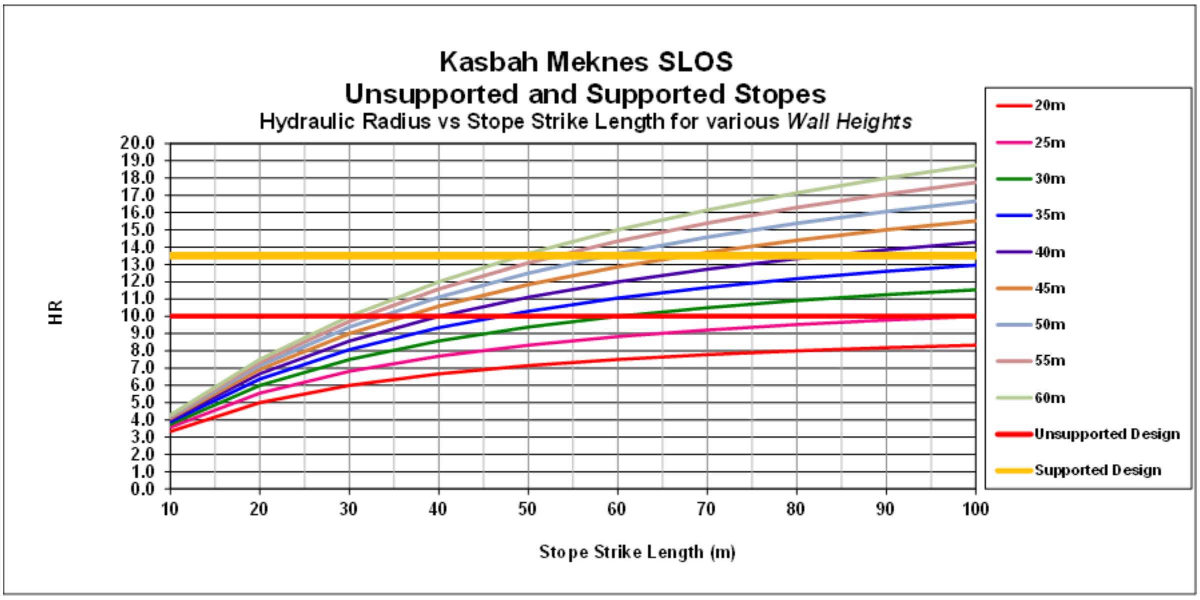


Figure 4-21 Meknès Zone-Design guidelines for stable unsupported and supported walls.

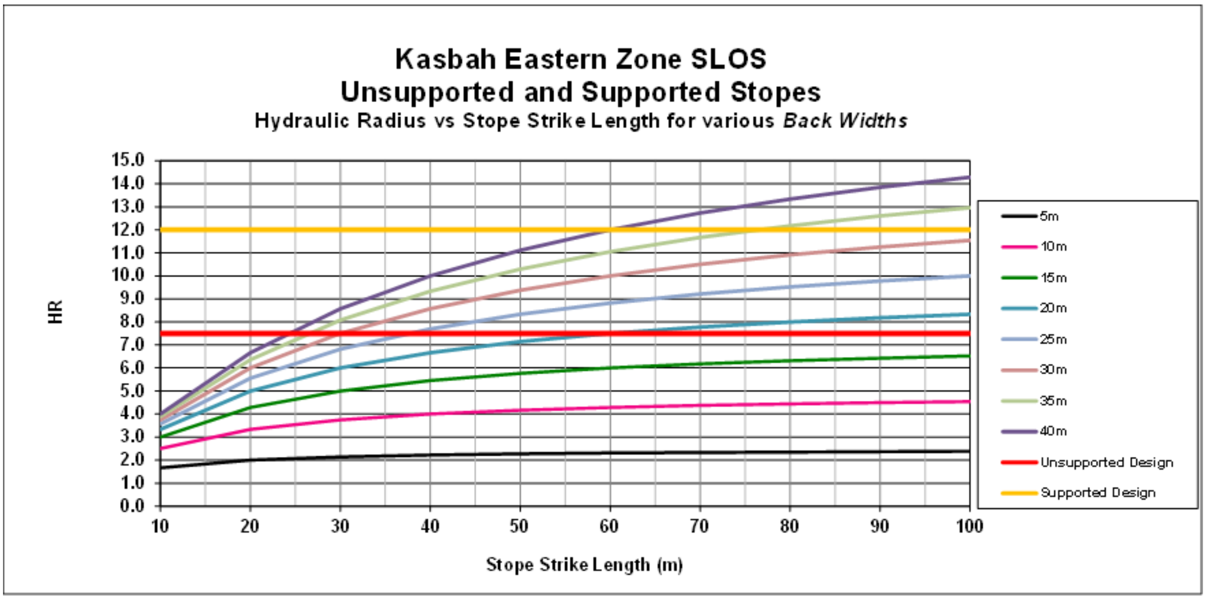


Figure 4-22 Eastern Zone-Design guidelines for stable unsupported and supported backs.

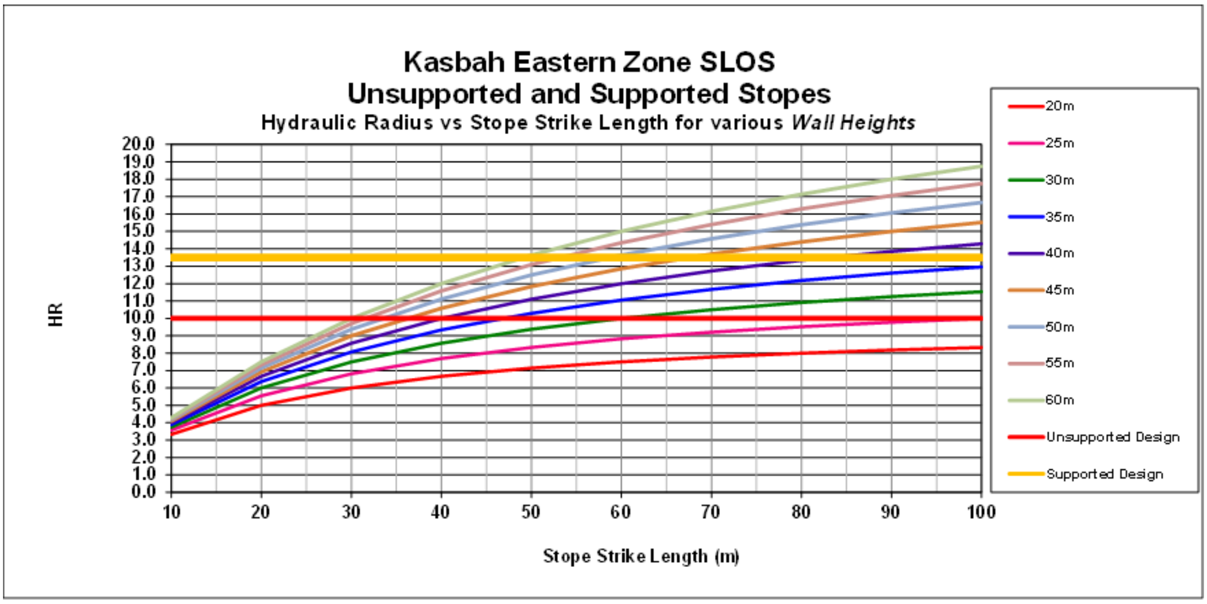


Figure 4-23 Eastern Zone-Design guidelines for stable unsupported and supported walls.

The initial evaluation indicated that various stable stope dimensions could be used to complete the mine designs. Some typical stable stope dimensions are outlined in Table 4-16 to Table 4-18.

Table 4-16 Back span estimates for the initial stope design purposes.

Width	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
9m	+100m	-	+100m	-
25m	-	30m	-	+100

Table 4-17 Stope height estimates for the initial stope design purposes inclination 55°

Height	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
30m	60m	60m	+100m	+100m
60m	30m	30m	50m	50m
90m	25m	25m	40m	38m

Table 4-18 Stope height estimates for the initial stope design purposes inclination 90°

Height	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
25m	90m	90m	+100m	+100m
50m	35m	35m	60m	60m
75m	27m	27m	42m	42m

Stope Pillars

A pillar layout has been proposed for the Eastern Zone where a top-down mine design will be implemented. Once a stable unsupported / supported HR was determined for the backs and walls of the stope, a stable pillar design guidance could be established. The design based on this guidance was modelled and tested for overall stability.

Solid support, in the form of pillars of ore or waste rock, is very commonly used to provide stability of excavations in many mining methods.

The design of pillars for support of mine openings requires the determination of two aspects:

- the pillar strengths
- the stresses acting on the pillars.

Once these two aspects are known, individual pillars and pillar layouts can be designed, according to the degree of stability required.

The method for calculating hard rock pillar strength that was originally proposed by Hedley and Grant (Hedley & Grant, 1972) was applied. The pillar strength equation is:

$$P_s = k \cdot \frac{w^{\alpha}}{h^{\beta}}$$

P=pillar strength

k=strength of 1m^3 of pillar material

w=pillar width

h=pillar height

α=0.5

β=0.75

It is also recommended that the value used for k is Laubscher's design rock mass strength (DRMS) (Laubscher, 1990). The pillar strength formula recommended for use in design and stability evaluation therefore becomes:

$$P_{strength} = DRMS.\ W_{eff}^{0.5}/H^{0.75}$$

Experience has shown that the DRMS usually falls in the range of 20% to 50% of the UCS (134 MPa) of the intact rock and is commonly about 30% of the UCS. This guidance suggests that the DRMS should be between 27 MPa and 67 MPa.

Alternatively, as part of the mining rock mass classification system, a rock mass strength (RMS) and design rock mass strength (DRMS) can be calculated (Laubscher, 1990). The RMS is determined from the UCS of the rock and the MRMR of the rock mass, using the relationship:

$$RMS = UCS \times (RMR - RUCS)/100$$

RUCS is the rating value in the MRMR table corresponding to the relevant rock UCS.

The DRMS is the rock mass strength in a specific mining environment. It is obtained by applying adjustments to the RMS to take into account the effects of weathering, joint orientation and method of excavation. (Table 4-19).

Table 4-19 DRMS values for Achmmach rock types.

	UCS	R-UCS	RMR	RMS	DRMS
Data perpendicular to bedding / foliation	134	10	65	74	56

A conservative DRMS value of 56 MPa, based on the average of the data perpendicular to the foliation has been used for the calculations.

The stresses, which will act on a pillar, will depend on the in-situ stress conditions and on the extent of both local and regional mining extraction. In regular mining layouts stresses in stope pillars are commonly determined using the tributary area theory.

This theory simply assumes that the pre-mining stresses are evenly redistributed through the pillars. The average pillar stress is the pre-mining stress distributed over the post-mining pillars. Therefore, if σv is the pre-mining vertical in situ stress, the average pillar stress P_{stress} in a sub-horizontal mining layout is given by:

$$P_{stress} = \sigma_v/(1 - e)$$

Where:

e is the extraction ratio (percentage extraction is 100 x e)

Once the pillar strength and pillar stress are known, the factor of safety (FoS) of the pillar can be determined as the ratio of the strength to stress:

$$\text{Pillar FoS} = P_{strength}/P_{stress}$$

An FoS of unity is equivalent to a probability of failure of 50%. The choice of the FoS value to be used for the design of the pillars and layout depends on the function of the pillars.

To account for the angle of orientation (θ) of the stress field relative to the design the average pillar stress was calculated using the following formulae:

$$Average\ Pillar\ Stress = \frac{(\sigma_V.\cos^2\theta)+(\sigma_h.\sin^2\theta)}{(1-e)}$$

Pillar Design Process

The following basic design parameters were used to determine the initial pillar specification:

- depths 50 m – 520 m
- ore body widths varying between 3 m – 12 m
- stope heights varying between 30 m – 90 m (on a 55° stope inclination) and 25 m – 75 m (on a vertical stope)
- stope Strike Lengths varying between 25 m and 60 m (on a 55° stope inclination) and 27 m – 90 m (on a vertical stope)
- pillar widths and lengths varying according to ore body widths and stope geometry
- host rock density (2.78/m^3)
- pillar strengths (DRMS 56 MPa based on a UCS of 134 MPa).

Factors that affect the design include:

- pillar width: height ratio (aspect ratio)
- factor of safety (FoS)
- pillar stress/UCS ratio.

For the purpose of providing design guidelines a minimum aspect ratio of 1:1 with a target of 1.5:1 was maintained while a factor of safety of 1.4 was targeted. The results were then compared with industry accepted norms for stable pillars (Pakalnis, 1994). The results relative to industry standards are presented in Figure 4-24. The pillar design guidelines are presented in Table 4-20 and Table 4-21.

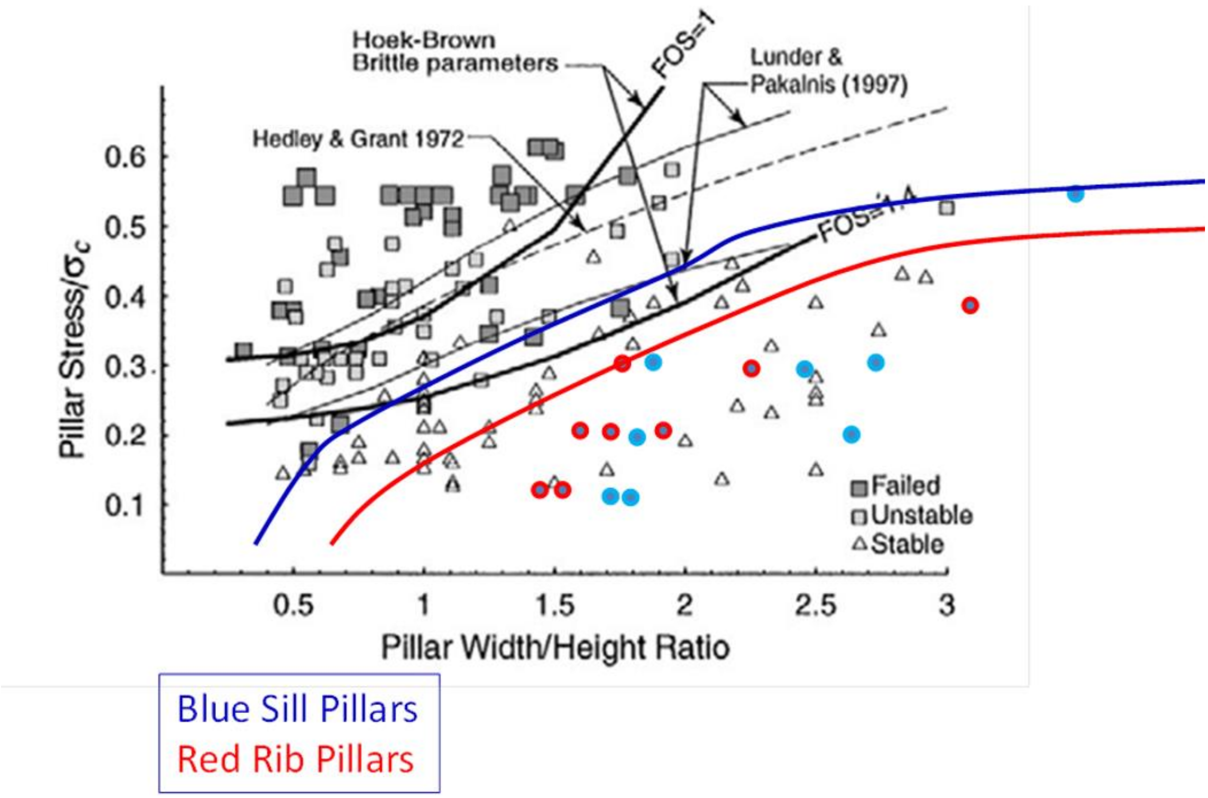


Figure 4-24 Comparison between Achmmach pillar and industry designs.

Table 4-20 Pillar design guidelines for the upper Eastern Zone.

Width	Sill		Rib	
	East	Meknès	East	Meknès
3m	3m	N/A	3m	N/A
6m	6m	N/A	6m	N/A
9m	9m	N/A	9m	N/A
12m	12m	N/A	12m	N/A

Table 4-21 Pillar design guidelines for the lower Eastern Zone

Width	Sill		Rib	
	East	Meknès	East	Meknès
3m	6m	N/A	7m	N/A
6m	9m	N/A	11m	N/A
9m	12m	N/A	14m	N/A
12m	15m	N/A	18	N/A

4.10.2 Void Stability (Dilution)

Over-break assumptions have been validated using hydraulic radius and modified stability number values and comparing these to dilution databases for mines with similar rock mass conditions (Mathews, et al., 1980).

As there is little variability in the rock mass characterisation across the strike of the mine the dilution does not need to be considered separately for different sections of the mine. Varying stope dimensions proposed for the four rock mass quality categories result in an equivalent linear overbreak/slough (ELOS) or dilution estimate less than 0.5 m. Results are presented in Figure 4-25 and Table 4-22.

Table 4-22 Equivalent Linear Over-break/Slough. Achmmach

	25% Quartile	Average	75% Quartile
East 25m * 90m	<0.5m	<0.5m	<0.5m
Meknès 30m * 60m	<0.5m	<0.5m	<0.5m

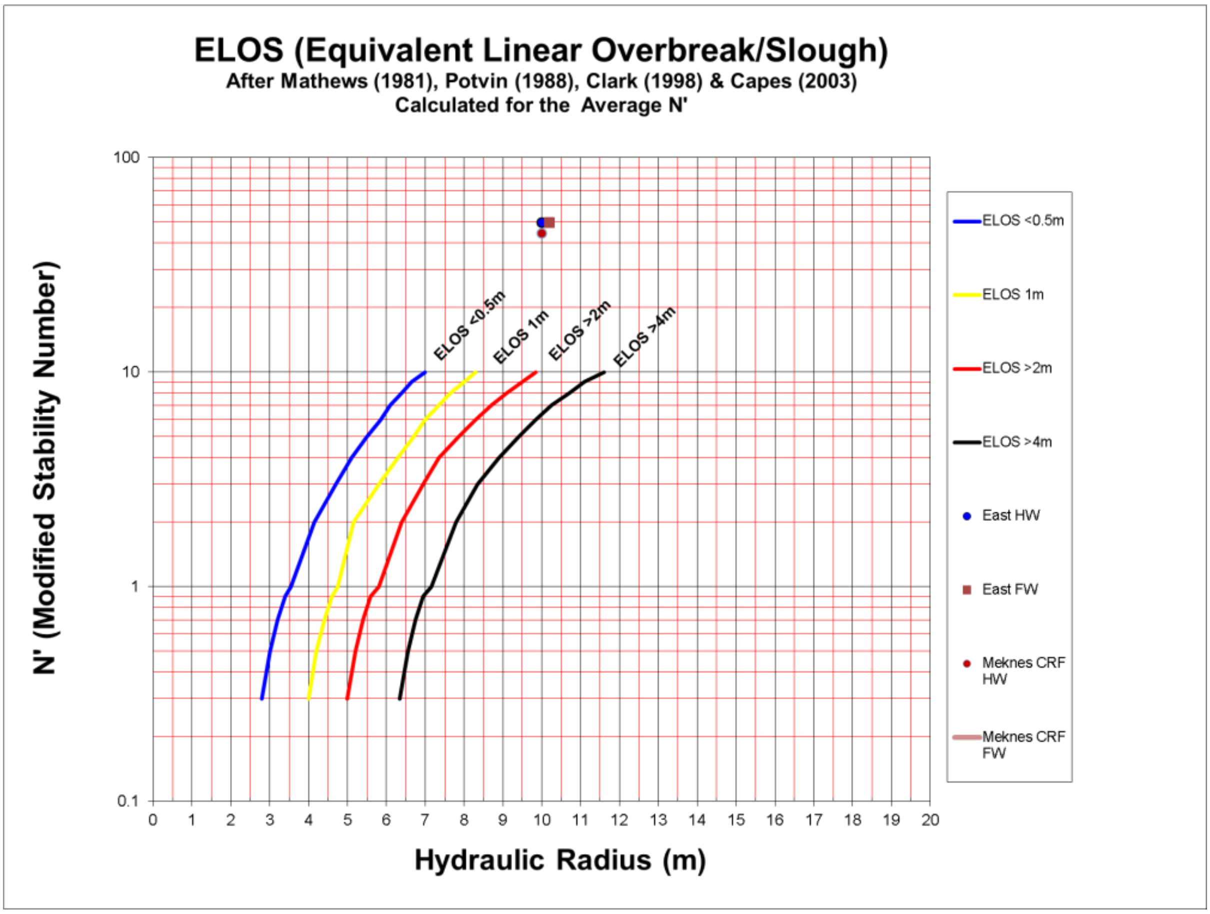


Figure 4-25 ELOS / Dilution graph for average Stability Numbers.

Given the expected low rates of dilution, additional in stope support such as cable bolts has not been considered.

4.10.3 Cemented Rock Fill (CRF)

Cemented Rock Fill (CRF) typically consists of mine waste rock or quarried aggregate that is mixed with a cement slurry (typically 3-10% by weight) and placed in an open stope. In order to fulfil its role as a passive support element and prevent excessive ore dilution CRF needs to be designed to maintain a stable wall 25 m high and up to 25 m wide. On the current bottom up plan the CRF will not be undercut and therefore does not need sufficient strength to be self-supported.

Due to the coarse-grained nature of CRF it is difficult to obtain representative geomechanical laboratory test results. Figure 4-26 provides comparative values from different mines of the strength to cement content of CRF. This provides a guide for design and costing purposes.

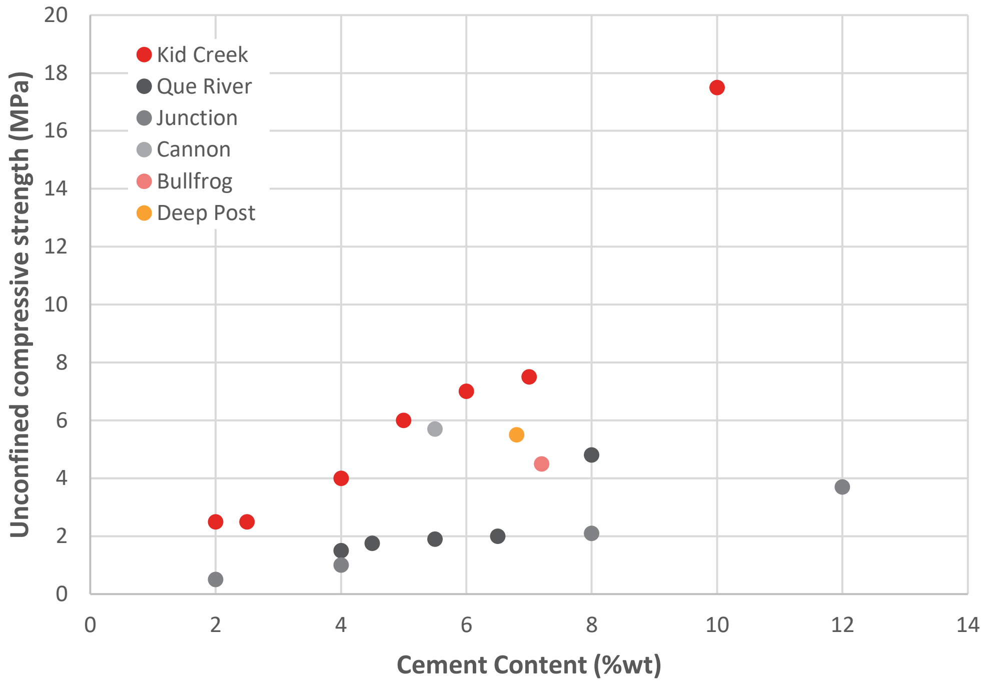


Figure 4-26 Examples of strength of cemented fill cf cement content.

Other factors that affect the strength of CRF material include:

- water to cement ratio
- aggregate strength
- water quality.

The most important factor for maintaining a properly engineered and well-consolidated CRF material is the need to minimise aggregate segregation during placement. Particle segregation has been observed in many CRF operations where large open stopes are filled by central filling using conveyors and single or centrally located fill raises. However, Stone (2007) reports that with the use of trucks to batch fill small stopes, high levels of CRF strength and minimal segregation effects can be maintained.

Several analytical and empirical techniques are available to assess the vertical and horizontal exposure of cemented fill masses. Mitchell, et al. (1982) developed a series of analytical solutions to assess the exposure of simplified vertical fill exposures. Mitchell and Roettger (1989) later developed a series of two-dimensional analytical solutions to the main failure modes of horizontal (or underhand) fill exposure.

Prior to any detailed analysis of backfill exposure stability at Achmmach, specific geomechanical laboratory testing on the representative fill material is recommended. In any case given the mining method and type of exposure (wall exposure) estimates based on historical results from other mines provide reasonable guidance for the current purposes.

Figure 4-27 presents the target unconfined compressive strength (UCS) required to maintain vertical exposure stability for stope heights between 20 and 60 m and exposure lengths between 10 m and 25 m. A factor of safety of 1.5 has been applied which is appropriate for a non-entry vertical exposure.

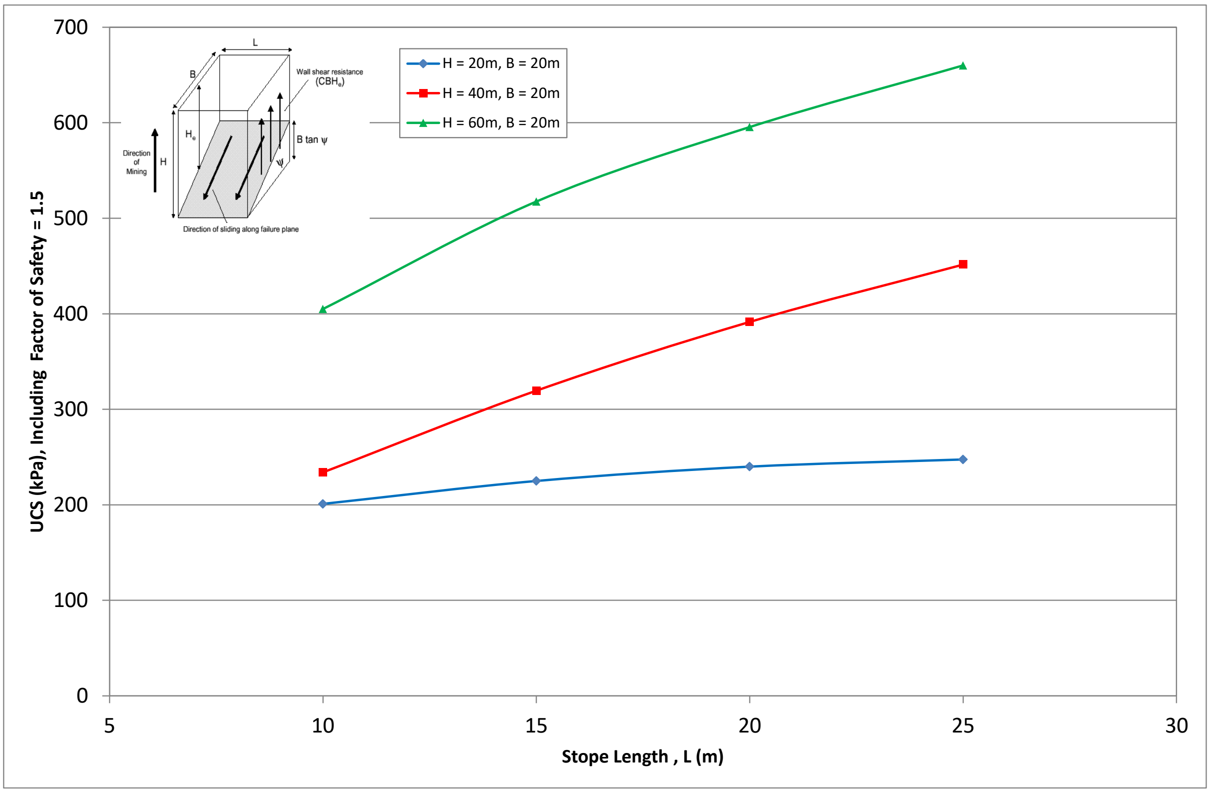


Figure 4-27 Analysis of CRF exposure stability.

Belem and Benzaazoua (2008) and Stone (1993) report that in the absence of numerical modelling, the two-dimensional limit equilibrium analyses typically result is an overly conservative estimate of limiting or critical strength, which increases backfill operational costs. In any case applying the design criteria indicates a required UCS of 250 kPa.

Modelling within continuum based numerical methods has been successfully employed by many practitioners to overcome the many assumptions, limitations and geometrical constraints associated with these analytical solutions (Caceres, 2005, Swan & Brummer, 2001). Connors, (2001) reports that numerical modelling of underhand CRF exposures at the Murray Mine in Nevada showed that a significant reduction in CRF cementation could be achieved. Given this it is expected that a design UCS of 200 kPa should be acceptable.

The relatively low UCS requirement of the CRF relative to what has been achieve elsewhere (Figure 4-26) suggests that a 5% cement content is adequate. Relatively straightforward screening on the rock fill material is also likely to increase the CRF strength and thus potentially reduce the cement content.

4.11 Mining Method Review

The Achmmach mine plan employs open stoping across the Eastern and Western zones, while using a modified Avoca method for the Central Zone. The Central Zone is targeted for early production predominantly because of its high grade.

4.11.1 Open Stoping

The proposed open stoping mining method is well-established and the good ground conditions and relatively narrow mining widths suggest that the Eastern Zone is well suited to this approach. The current geotechnical analysis validates the open stope mine design.

4.11.2 Modified Avoca with Cemented Rock Fill (CRF)

The Avoca method is typically used to maximise the recovery of a resource by utilising an unconsolidated fill (fill without a binder such as cement). The central idea is to advance a rock filling face with the stoping front such that the exposed stope walls do not exceed the stable hydraulic radius. As a result, few if any pillars are required. The high resource recovery rates and low backfilling cost are usually offset by the fact that additional capital development is required to establish backfilling access on the relevant level. Figure 4-28 is a schematic of the Avoca method.

The additional capital cost associated with the Avoca method has typically motivated mine planners to develop modified versions. For Achmmach it is proposed the use cemented rock fill instead of loose rock fill in part to minimise the capital expense. The use of CRF allows a single access to be used for both production drilling and backfilling. As a result, the mining will be more cyclic, though this disadvantage is offset by savings through reduced capital development.

The Avoca method is bottom up and any CRF pillar is required to be of sufficient strength that it can maintain a vertical face over a one level interval. The cement content is therefore expected to be minimal (see section 3.3 above). To minimise cement content further, only the first half of a 60m stope needs to be filled with CRF (shown in green in Figure 4-29) and the remainder of the void would be filled with loose rock-fill.

The modified Avoca method proposed for the Central zone is complicated by the fact that the ore is quite wide in this area. If re-entry is required for backfilling or ore production then the stope design will need to be profiled so as to form stable arch as shown in Figure 4-30. This will result in some ore loss. Additional ground support (such as cable bolting) may also be required, although the current assessment indicates that if the profiling of the stope and back filling is effective then then re-entry will be possible following some rehabilitation (scaling and spot bolting).

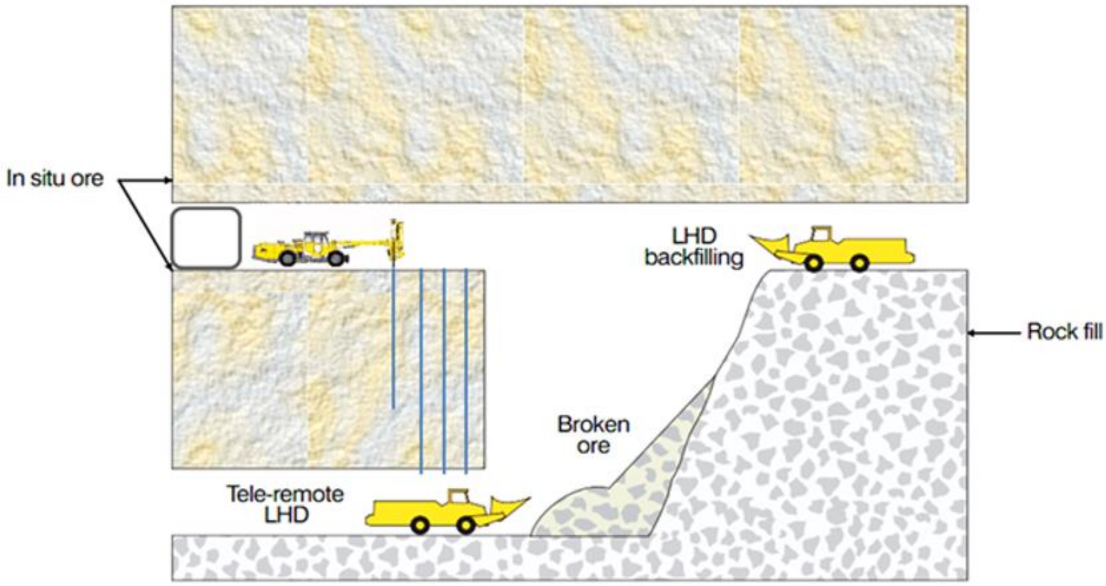


Figure 4-28 Schematic showing typical Avoca method.

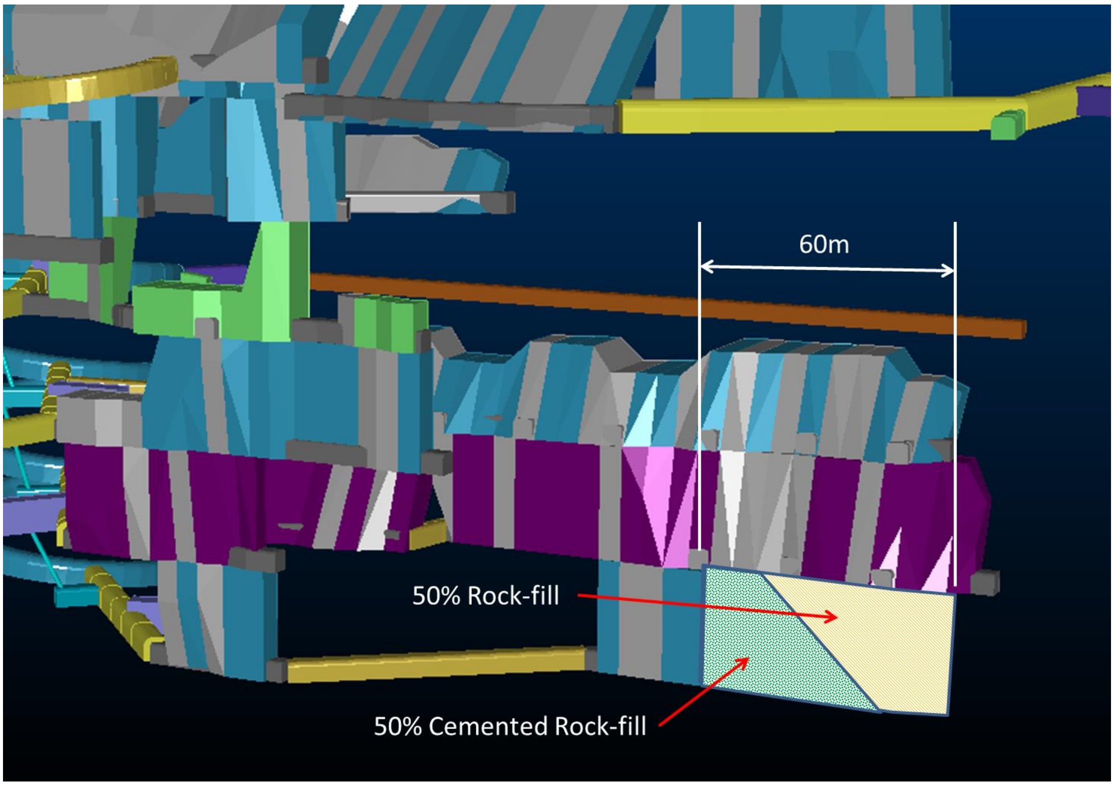


Figure 4-29 Modified Avoca using CRF.

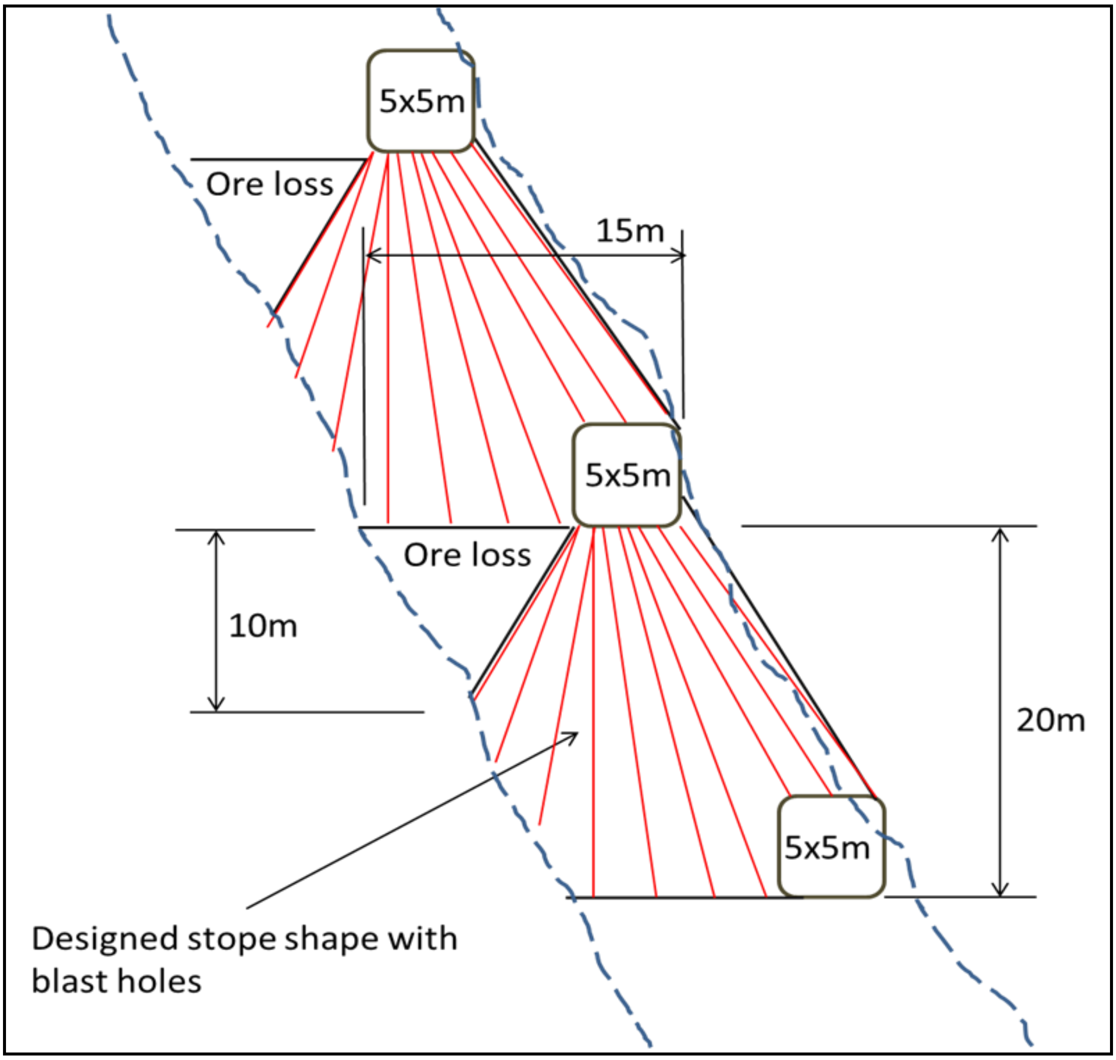


Figure 4-30 Cross-sectional view of stope design when using Avoca for wide orebodies.

Key to the success of profiling the stope will be utilising down-holes. This will help maintain the stability of the profile of the stope while minimising damage to ground support.

4.12 Mine Stability Modelling

Numerical models of the mine and geotechnical conditions were constructed and analysed using Flac3D, a three dimensional boundary element package. The numerical models contained the current planned life of mine (LoM) development and stope wireframes.

The main objective of the numerical modelling was to determine the pillar stresses in the Eastern Zone and thus validate or otherwise the design criteria used. The model was also used to identify any potential stress change influence on the declines as stope extraction progresses.

4.12.1 Model Parameters

Geometry

The modelled mine geometry and mining sequence was input to the model. Figure 4-31 is a longitudinal section of the Achmmach areas.

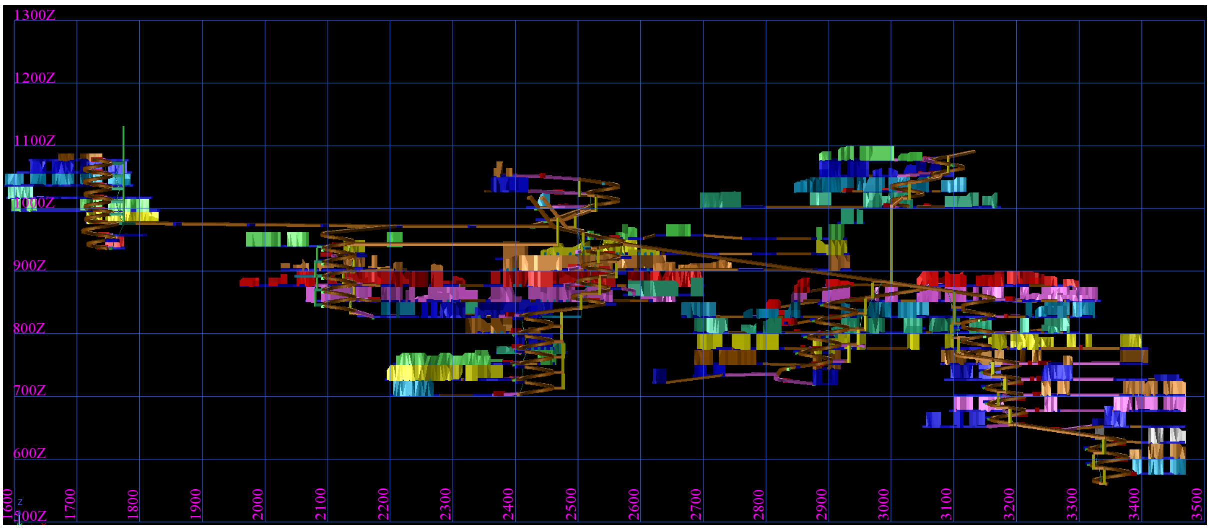


Figure 4-31 Longitudinal Section of the Achmmach Project provided for Flac 3D Model

In-Situ Stress Field

The stress field adopted for the modelling was derived from the stress estimates described in the previous section and shown in Table 4-23.

Table 4-23 Stress field Meknès and East Zones

Stress Magnitude	Trend	Plunge
σ 1 = σ 2 * 1.1	150°	15°
σ 2 = 0.027 * d	0°	75°
σ 3 = σ 2 * 0.9	60°	15°

Material Properties

The material parameters selected for the model were based on the laboratory test results and rock property analysis. Rocscience's RocLab software package was used to determine the rock mass parameters from the laboratory test results.

Young's Modulus was derived from well-known relationships available in the literature. The rock mass Young's modulus was estimated from the Hoek and Dietrich's relationship (Hoek & Diederichs, 2006).

$$E_{rm} = E_i \left(0.02 + \frac{1 - D/2}{1 + e^{\frac{60+15D-GSI}{11}}} \right)$$

The Poisson's ratio was estimated using the expression proposed by (Flores & Karzulovic, 2003):

$$\nu = 0.4 - \frac{GSI^{0.7}}{100}$$

No laboratory tests were available for the Intact Modulus (Ei) of the orebody and waste rock. Values for the waste rock were assumed at 30 GPa and for the orebody 75 GPa. The modelling rock mass parameters assigned to the host material for Flac 3D numerical modelling are described in Table 4-24.

Table 4-24 Rock Mass Parameters

	D	UCS (Mpa)	GSI	Erm (GPa)	v	C (kPa)	Friction (°)
Waste Rock Mass	0.0	75	58	14	0.23	1995	48
Orebody Rock Mass	0.0	134	59	37	0.23	2801	55

4.12.2 Numerical Model Results

Regional Analysis

The major principal stress acting horizontally in a northern direction, results in a stress concentration at the abutments of the planned stoping. As the stoping foot print increases, the induced stress field increases, and typical low tensile zones are created in the hanging wall and footwall of the stoped out area. The analysis shows that the overall regional stability of the mine is satisfactory.

The numerical modelling does not include cemented rock fill (CRF) material due to a minimal potential influence on regional stability and local stope stability. Not including CRF material in the model provides a worst-case analysis. The inclusion of CRF in the numerical modelling may lead to erroneous results due to the uncertainty of the assumptions required for the fill material properties.

Declines

The main declines providing access to the various stoping areas are in the foot wall of the mineralised zone and are positioned to minimise the length of the required access development. The numerical modelling indicates that none of the declines will be influenced by stoping activities throughout the life of the mine to a degree that will cause failure of the rock mass through stress. Two key factors that play a role in this are the shallow depth of the mine and the stoping stress shadow that will be cast beyond the proposed locations of the declines.

Stoping and Pillars

The analysis showed the potential for medium stress levels to form in the pillars left between multiple lenses that are stacked closely with one another. Although it is not believed that this will affect the overall global stability of the surrounding ore drives or accesses, it may have the potential to cause localised failure in the hanging wall resulting in over break. Should this occur it expected to be able to be managed by additional ground support in the affected areas.

In summary Flac 3D modelling results show that:

- Low tensile zones are present in the hanging and footwall of the stopes and overbreak is expected to be <0.5m in the footwall and hanging wall.
- Pillar stresses are in line with the Hedley and Grant methods.
- The results from the modelling compares well with industry data.

4.13 Box Cut and Portal

An improvement in the understanding of the ore body and subsequent mining strategies indicated that two declines would maximise the production profile at the Achmmach project area. Box cuts, portal entrances and declines were therefore planned at the Central and Eastern locations (Figure 4-32).

Geotechnical holes were drilled to obtain additional geotechnical information for the horizons in which the box cuts, portal and declines will be developed.

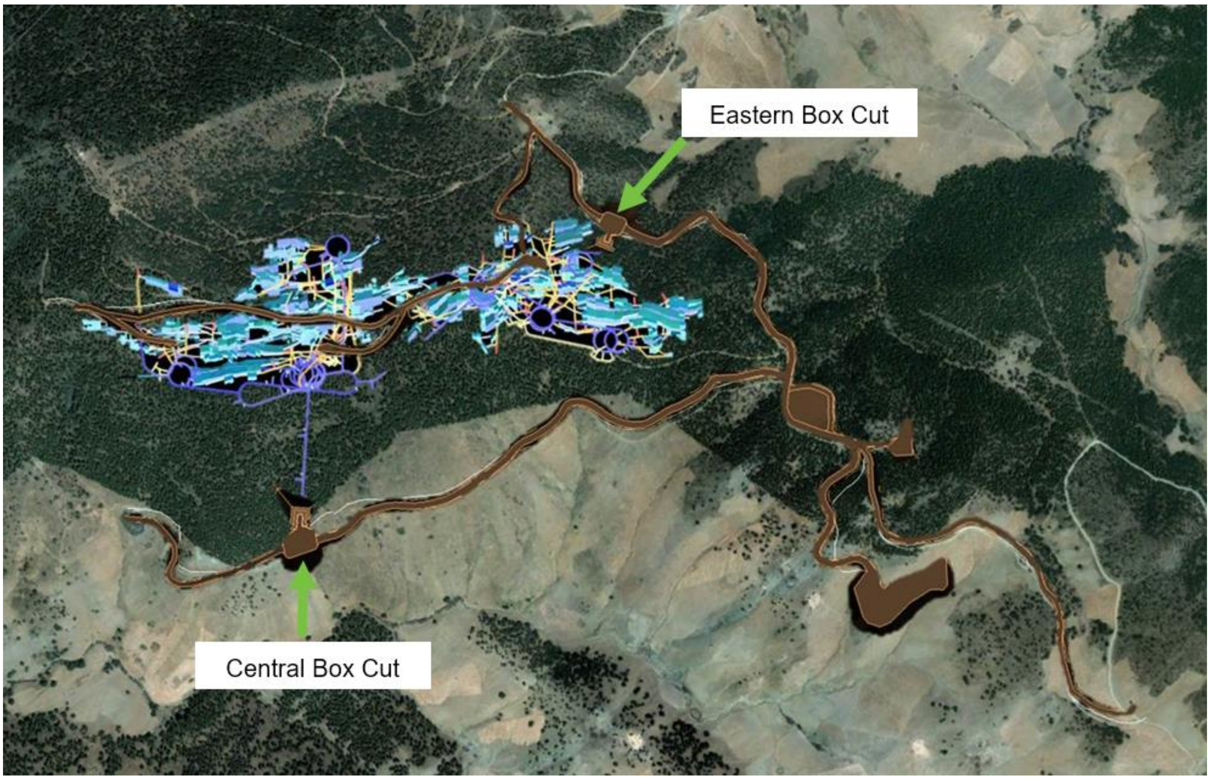


Figure 4-32 Surface layout showing central and eastern box cut positions

The general area in which the box cuts will be developed is hilly (slopes of ±30°) and is covered with trees and grass.

Three types of slopes will be formed in the Central and Eastern Box Cuts. These are (1) soil / scree, (2) moderately weathered shale / siltstone / sandstone slopes and (3) jointed fresh hard rock slopes. The overall slope angles in the jointed rock mass were determined using the adjusted MRMR rock mass classification, appropriate design charts, computer modelling and experience in similar rock types.

The mechanical properties for the soil and moderately weathered material, obtained from the Rocscience RocLab data, were used together with design charts to determine the optimum slope angle and slope height in the weathered material.

4.13.1 Geotechnical Horizons

Three geotechnical horizons have been identified at Achmmach. These include a top layer of soil and scree a middle layer of moderately to slightly weathered rock and a base layer of fresh rock.

The contacts between the various levels geotechnical horizons are gradational. The summary, weathered depth metres below surface of the bore holes are presented in (Table 4-25).

Table 4-25 Summary of bore hole logs (depth below surface in metres)

	Central	Eastern	Design
Soil/Scree	0 – 5.0 m	0-2.0 m	0-4.0 m
Weathered Rock	5.0m – 12.0 m	2.0 – 12.0 m	4.0-12.0 m
Fresh	+12.0 m	+12.0 m	+12.0 m

Soil: Transported soil and rock
Weathered Rock: Moderately weathered shale / siltstone / sandstone
Fresh: Fresh shale, siltstones and sandstone with occasional weathered joint and / or bedding parting planes)

4.13.2 Design

Soil / Scree

The soil will be machined back to a height of 4.0 m and a batter angle of 45°.

Weathered Rock

The weathered rock slopes were analysed using the SLIDE computer modelling program. The method utilises the cohesion, angle of internal friction and the relative density of the material with the proposed height and a factor of safety of 1.5 for the slope.

A single 8 m high bench is proposed for the weathered rock. The batter angles for the benches will be 60°. A 4.0 m wide berm is required at the weathered rock and soil/scree boundary, as well as between the weathered and fresh hard rock batter.

Hard Rock

A single 10m high batter is proposed for the fresh rock types at Achmmach.

It is standard practice to design a boxcut to be sufficiently deep to ensure there is 5.0 m of fresh or only slightly weathered rock to form a suitable stable beam to create a stable portal entrance. A box cut with a depth of 22 m would be required for this scenario (Figure 4-33).

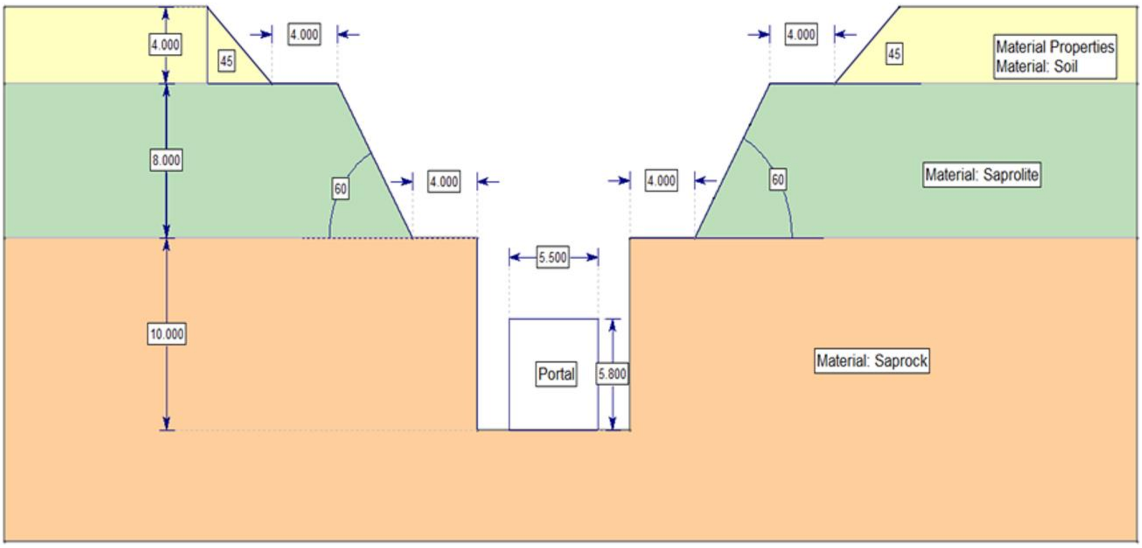


Figure 4-33 Schematic section of the boxcut (central and east)

A summary of the design is presented in Table 4-26.

Table 4-26 Design parameters for the boxcut

	Central Box Cut	East Box Cut
Soil	4m bench at 45° 4m berm	4m bench at 45° 4m berm
Saprolite	8m bench at 60° 4m berm	8m bench at 60° 4m berm
Fresh Rock	10m bench at 90°	10m bench at 90°
Portal	5.5m * 5.8m decline 5m solid beam ± 1 – 2m from the box cut sidewall	5.5m * 5.8m decline 5m solid beam ± 1 – 2m from the box cut sidewall

4.13.3 Kinematic Stability

The presence of joints in a rock mass, which has been exposed by a slope (box cut), may result in failure along discontinuities. Three primary modes of failure could occur in hard rock, namely plane, toppling, and wedge failure.

Plane Failure

No plane failure, associated with the prominent bedding, is expected in the portal high walls of the East and Central box cuts (Figure 4-35).

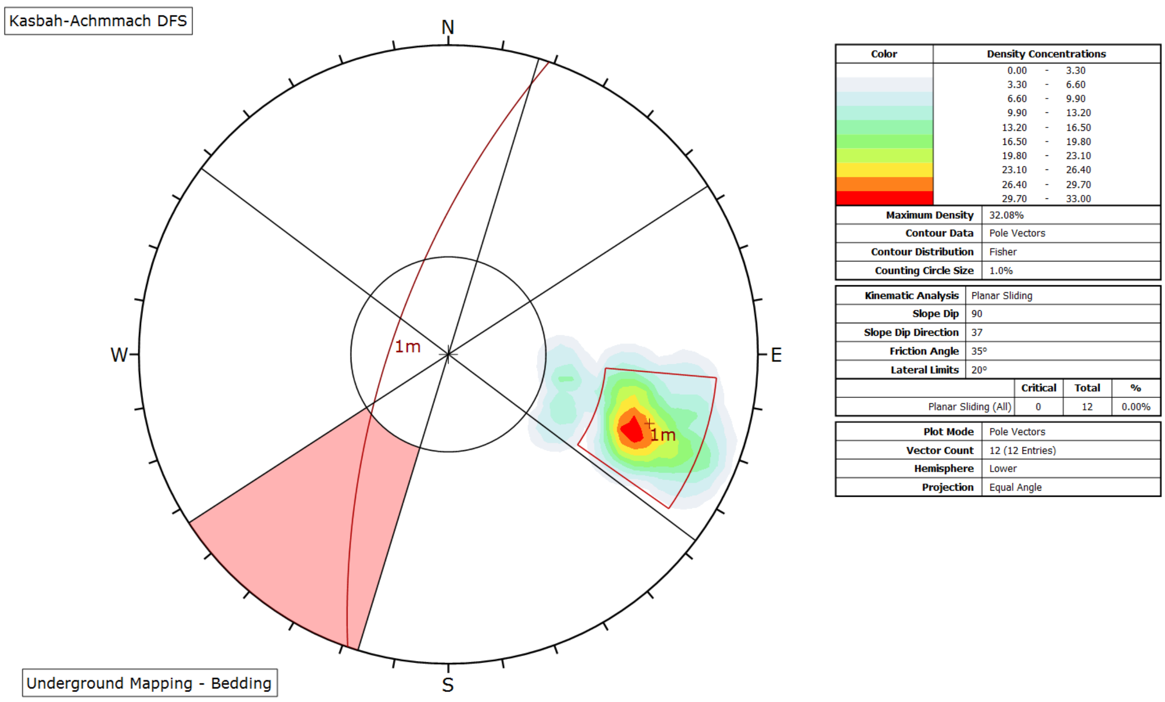


Figure 4-34 Plane failure in the portal high wall east boxcut

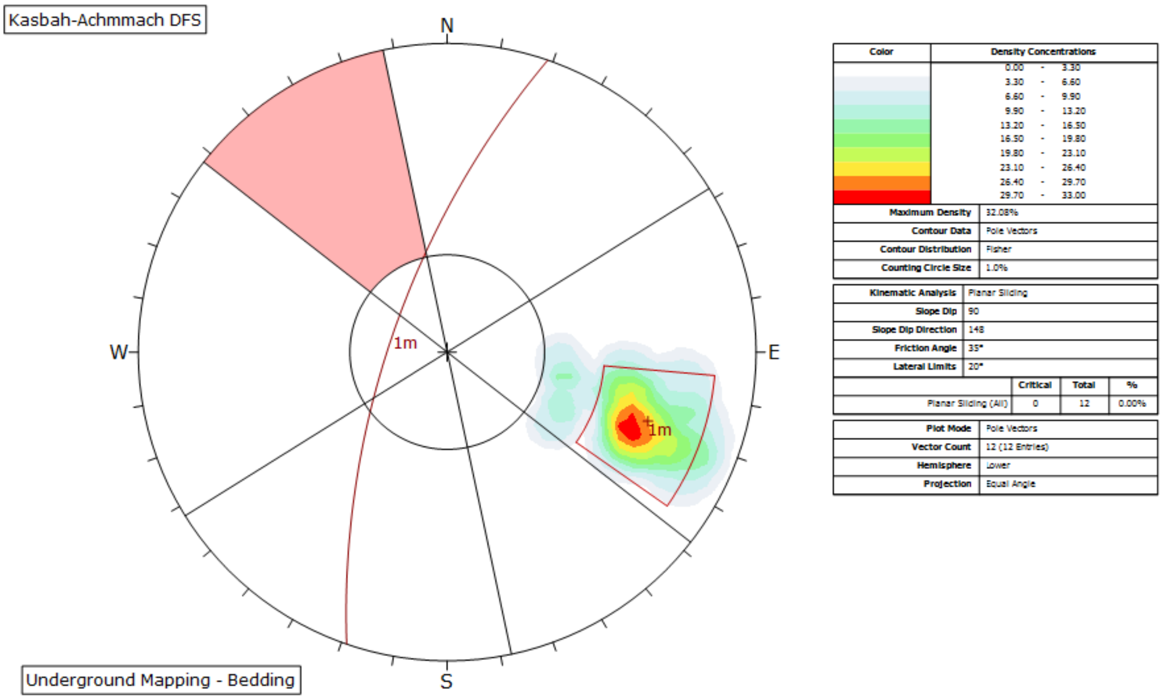


Figure 4-35 Plane failure in the portal high wall central boxcut

Plane failure could occur (east boxcut -10% possibility; central boxcut –8% possibility) along some of the inconsistent east–west striking (J1 and J3) joint patterns. These joints will be identified and mapped during the excavation process and evaluated and supported accordingly.

Toppling Failure

No toppling failure, associated with the prominent bedding, is expected in the portal high walls of the east and central box cuts (Figure 4-37).

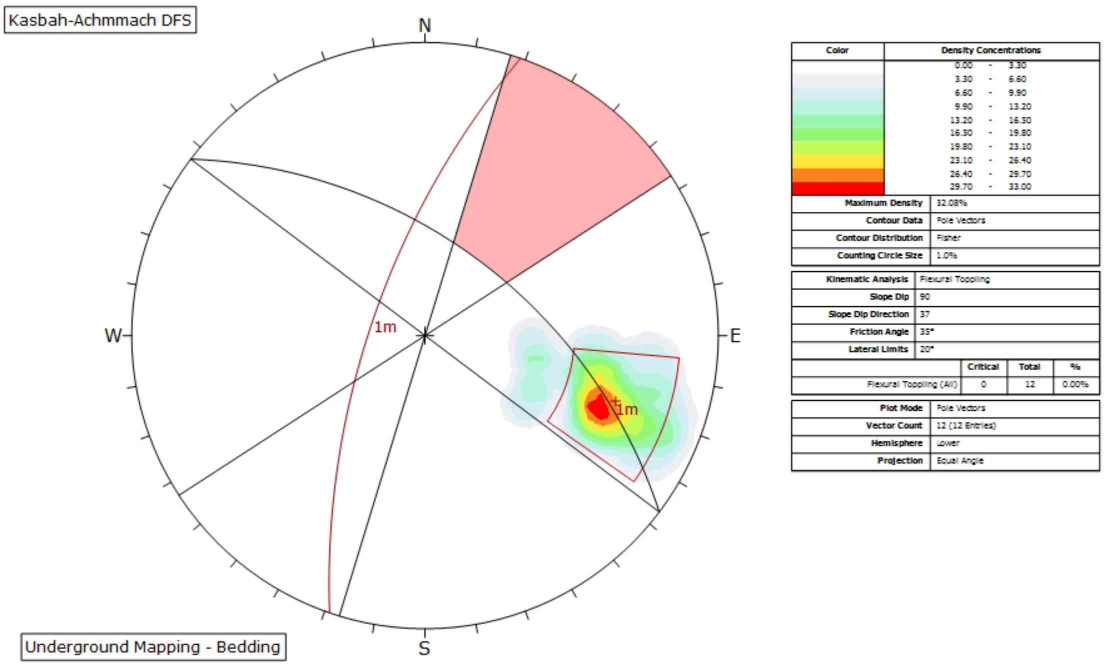


Figure 4-36 Toppling failure in the portal high wall east boxcut

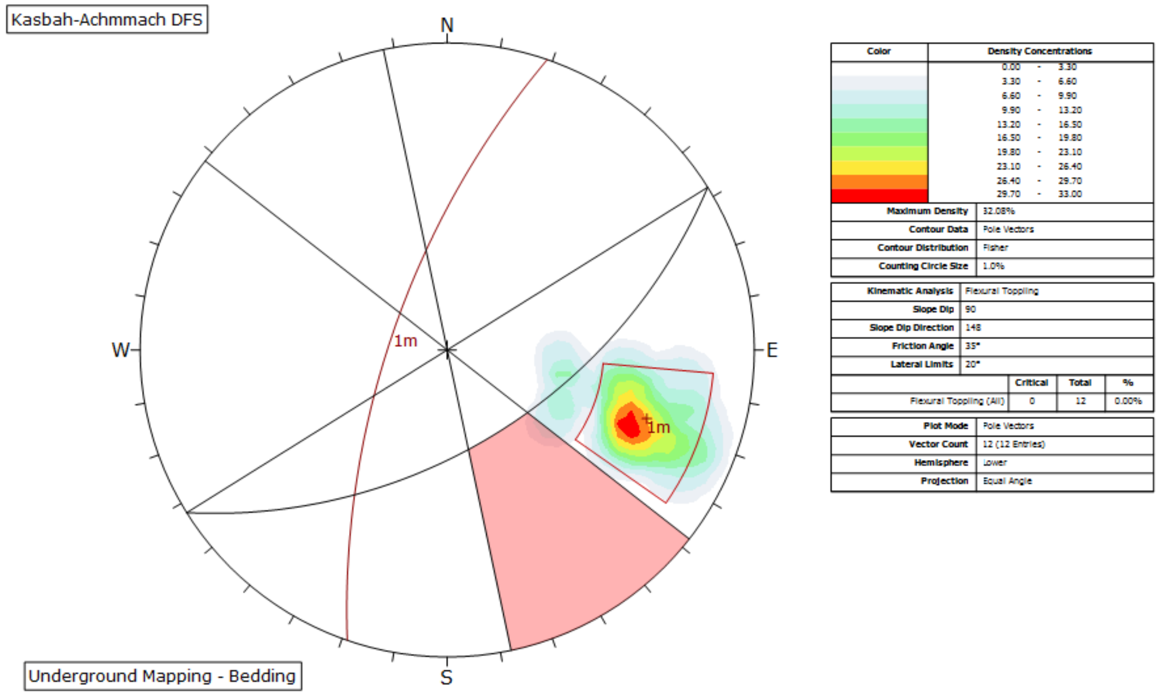


Figure 4-37 Toppling failure in the portal high wall central boxcuts

Toppling failure could occur (East Box Cut-18% possibility; Central Box Cut – 26% possibility) along some of the inconsistent east – west striking (J1, J3 and J4) joint patterns. These joints will be identified and mapped during the excavation process and evaluated and supported accordingly.

Wedge Failure

Wedge failure (possibility 20%) could occur in the portal high wall (Figure 4-39) along combinations of the joint orientations and the prominent bedding.

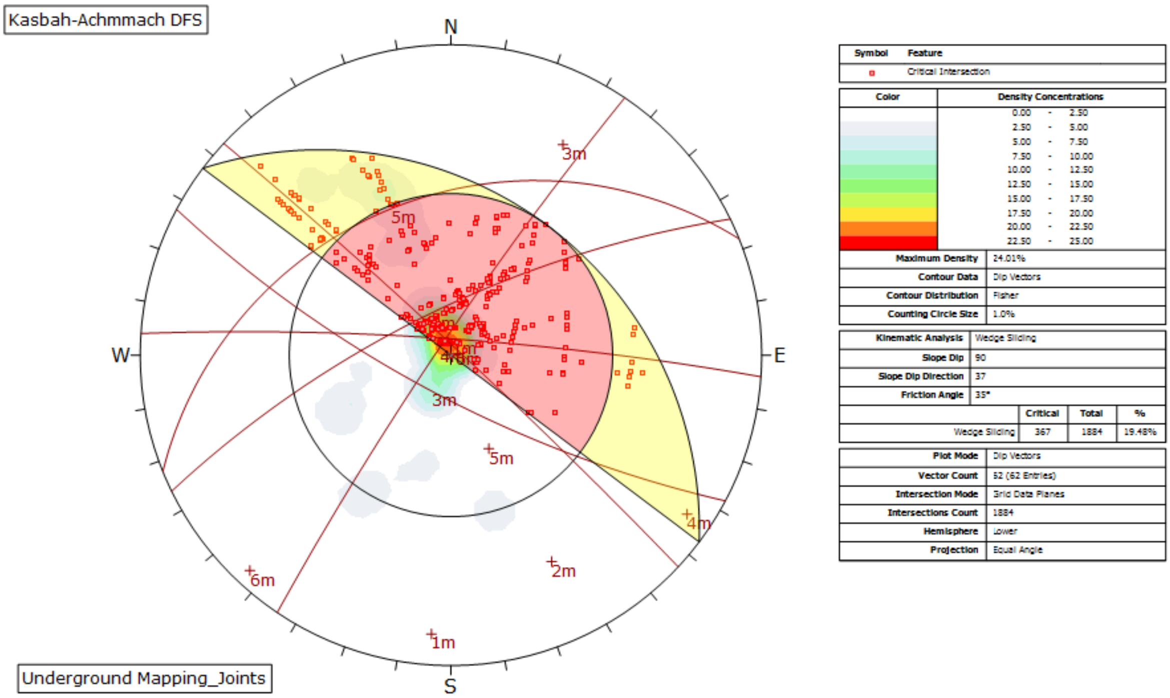


Figure 4-38 Wedge failure in the portal high wall east

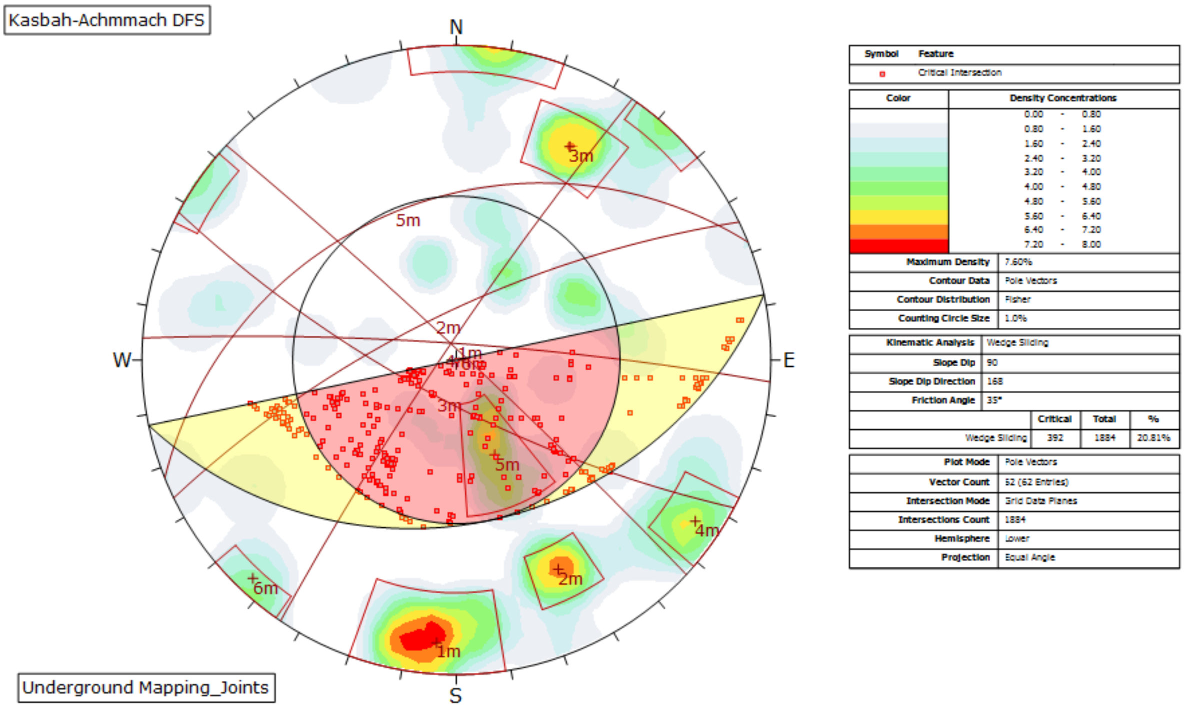


Figure 4-39 Wedge failure in the portal high wall west

These wedges will be identified and mapped during the excavation process and evaluated and supported accordingly.

SLIDE Computer Modelling

For purposes of quantifying the overall slope stability, the SLIDE numerical modelling software was used. The software can identify the most vulnerable circular surface for failure and evaluates the critical slip surface in terms of its factor of safety.

Given the box cut slope angles, a composite slope geometry was compiled. The purpose of this SLIDE model is to test the design geometry and parameters for stability (Figure 4-40). An acceptable factor of safety (1.55) is achieved according to the model.

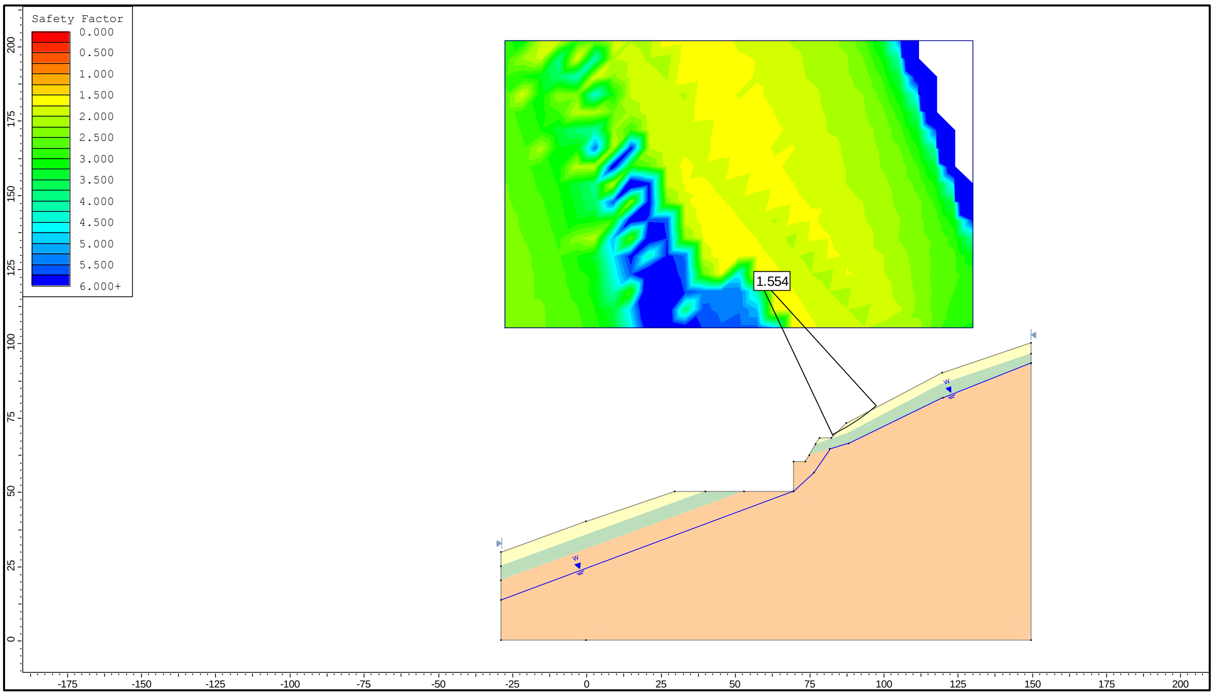


Figure 4-40 SLIDE model for the central and east box cut (FoS = 1.55)

4.13.4 Boxcut and Portal Support Strategies

Upper Soil Bench

- 4 m high and worked back at 45°. A 4 m wide berm at the toe of the bench
- 50 mm shotcrete (to be applied in 2 layers of 25 mm each)
- the first layer of shotcrete must be applied asap after being excavated
- galvanised mesh with apertures 75 – 100 mm; (do not use less than 50 mm because the shotcrete will not penetrate properly)
- the second layer of shotcrete to be applied over the first layer of shotcrete and mesh.

Middle Weathered Rock Bench

- 8 m high bench at 60° bench angle. A 4 m wide berm at the toe of bench
- 50 mm shotcrete (to be applied in 2 layers of 25 mm each)
- the first layer of shotcrete must be applied asap after being excavated
- galvanised mesh with apertures 75 – 100 mm; (do not use less than 50 mm because the shotcrete will not penetrate properly)
- the second layer of shotcrete to be applied over the first layer of shotcrete and mesh.

Lower Fresh Hard Rock Bench

- a single 10 m high bench a 90° angle
- 50 mm shotcrete (to be applied in 2 layers of 25 mm thick layers)
- the first layer must be applied asap due to the mechanical nature of the shale / siltstone / sandstone
- galvanised mesh between the two layers of shotcrete
- splitsets: 2.4 m (47 mm) long split sets on a 2 m * 2 m box pattern (Use the split sets / plates to pin the mesh).

Portal

Fully grouted Mechanical Long Anchors (6m long x 38t tensioned to 20t) installed in 2 rows. The 1st row to be installed 1.5m above the top contact of the decline and the 2nd row to be installed 2m above the 1st row. The Mechanical Long Anchors to be spaced 2m apart. Mechanical anchors could also be installed as trusses to improve clamping forces in blocky and jointed areas.

4.13.5 Box Cut Strategies

Blasting

Any blasting below the weathered rock will be done with care in order to limit any blast related damage. Control blasting techniques e.g. pre-splitting, must be done before the production blasts are done in order to create stable conditions with minimal blast damage. Four-metre-high benches will be blasted to minimise blast related damage.

Alternative Support

Alternative support types will be evaluated during the excavation:

- Resin injection could be required depending on the competency of the rock mass surrounding the portal.
- Steel arch sets or shotcrete arches could be required depending on the competency of the rock mass and decline stability.

The selected contractor will have the necessary tools and materials to install these alternative support types if required.

Water Management

Drainage around the box cut perimeter will be established to divert any stormwater away from entering the box cut.

Drain pipes +- 150mm in length will be installed to effectively drain the water behind the shotcrete.

Water monitoring holes will be drilled around the box cut and used to monitor groundwater levels. If water levels are above the floor of the ramp, the holes can be pumped to drain water away from the boxcut.

4.14 Development: Declines and Drives

Rock mass classification systems provide a convenient method of determining support requirements for excavation. The Q and RMR systems have been used to prepare the design charts, which show the required support types including rock bolt length, support density and shotcrete thickness. These charts are applicable for long term tunnels and chambers in reasonable rock and in situ stress conditions.

Two design methods were used e.g. (1) Barton (Civil Engineering approach) and (2) Stacey (Mining approach) as well as practical experience from various mines in similar rock types.

Development Sizes:

- main declines: 5.5 m wide and 5.8 m high (40% of all development)
- hangingwall: ore and footwall drives; 4.5 m wide and 4.5 m high (50% of all development).

4.14.1 Rockbolt Length

The required length of rock bolts is a function of the dimensions of the opening.

The conservative civil engineering approach indicates in the absence of statistical data reflecting the height of rock requiring support, the formula suggested by Barton or Brady and Brown for calculating the required support are as follows:

Support Length = (2 + 0.15 x span)/ESR

ESR-safety factor relating to the importance of the opening

Typically, the ESR for permanent mine openings will vary between 1.6 and 2. A conservative ESR value of 1.6 was used.

Table 4-27 Support length as per Barton (Q ratings indicates average conditions)

	Support Length
Declines (5.5m x 5.8m)	1.8 m
Drives (4.5m x 4.5m)	1.7 m

As a rule of thumb, for reasonable rock conditions, the length of the bolts should be a third of the span or the wall height of the excavation. In poor conditions the factor should be 0.5 and in good conditions 0.25 (Table 4-28).

Table 4-28 Support length as per Stacey (RMR indicates average conditions)

	Good	Average	Poor
Declines (5.5m x 5.8m)	1.4m	1.8m	2.8m
Drives (4.5m x 4.5m)	1.1m	1.5m	2.3m

4.14.2 Support Requirements

Both the Civil Engineering and Mining approaches utilise design charts as indicated in Figure 4-41 to Figure 4-43 and Table 4-29 and Table 4-30.

Civil Engineering Approach

The Civil Engineering approach utilises Barton's Q (Ref.10) to determine bolt spacing and shotcrete thickness.

Figure 4-41 is an indication of the support required for the 25%-50%-75% range of the Q-rating. Similar methods were followed for all types of rock masses and excavation dimensions.

The results are tabulated in Table 4-29.

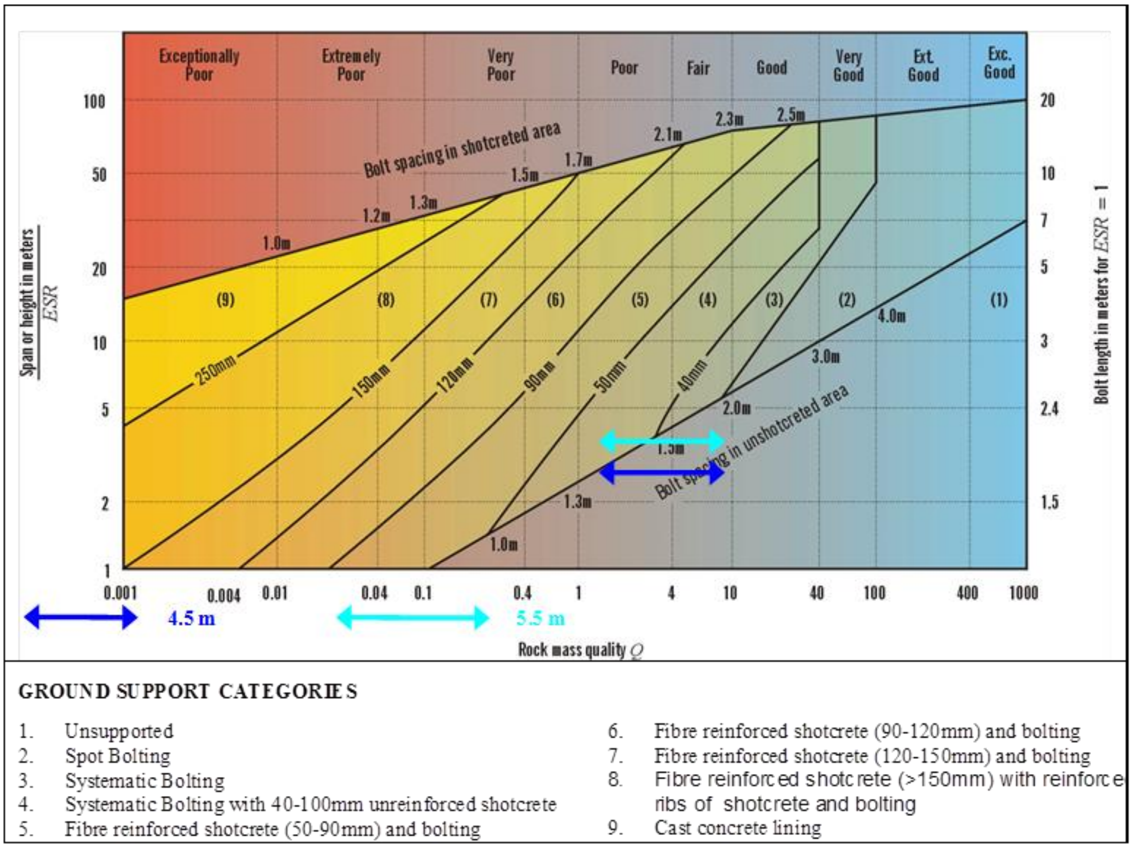


Figure 4-41 Civil engineering approach (based on Q)

Table 4-29 Support requirements as per civil engineering approach

	Rock Bolts	Shotcrete
Declines (5.5m x 5.8m)	Length 1.8 m Spacing 1.4 m-2.0 m	0 – 45 mm
Drives (4.5m x 4.5m)	Length 1.7 m Spacing 1.4 m-2.0 m	0 mm

Mining Approach

The mining approach from Stacey utilises the RMR criterion (Ref. 11) to determine bolt spacing and shotcrete thickness.

Figure 4-42 is an indication that no support to one bolt every 3 m^2 is required and Figure 4-43 is an indication that no shotcrete is required.

The results are tabulated in Table 4-30.

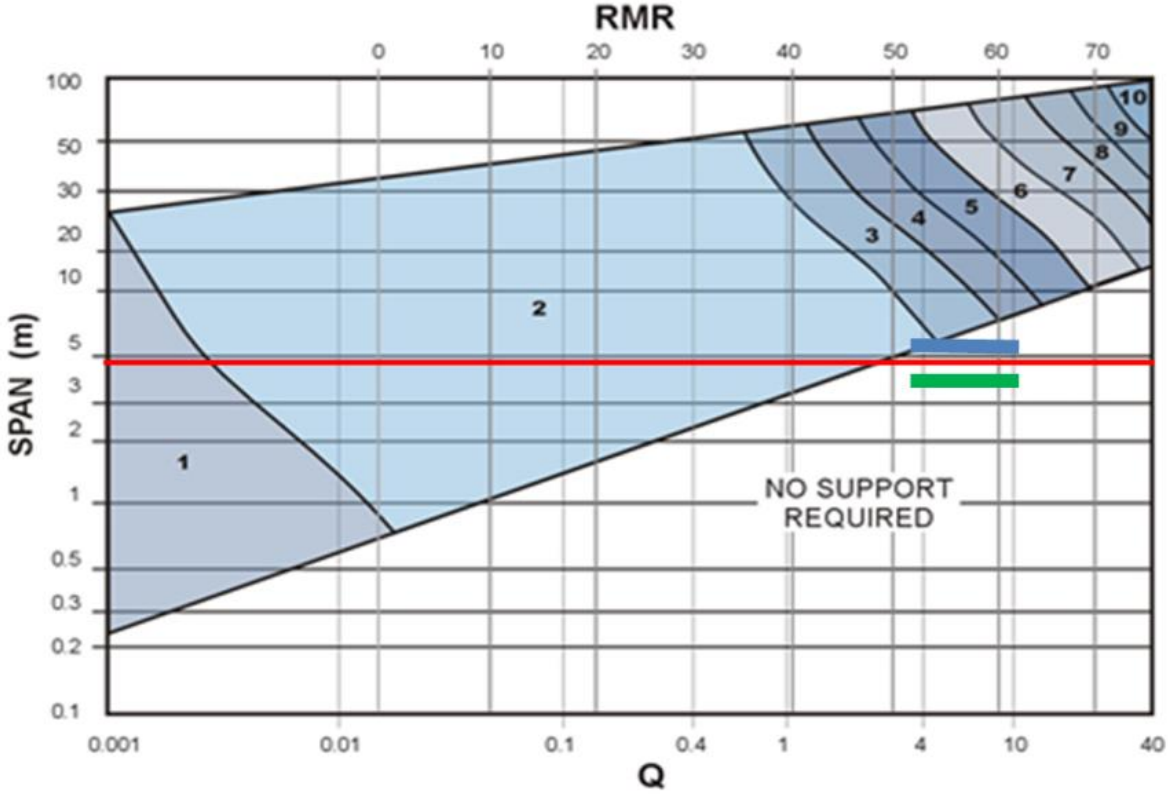


Figure 4-42 Mining approach for rock bolts (based on RMR)

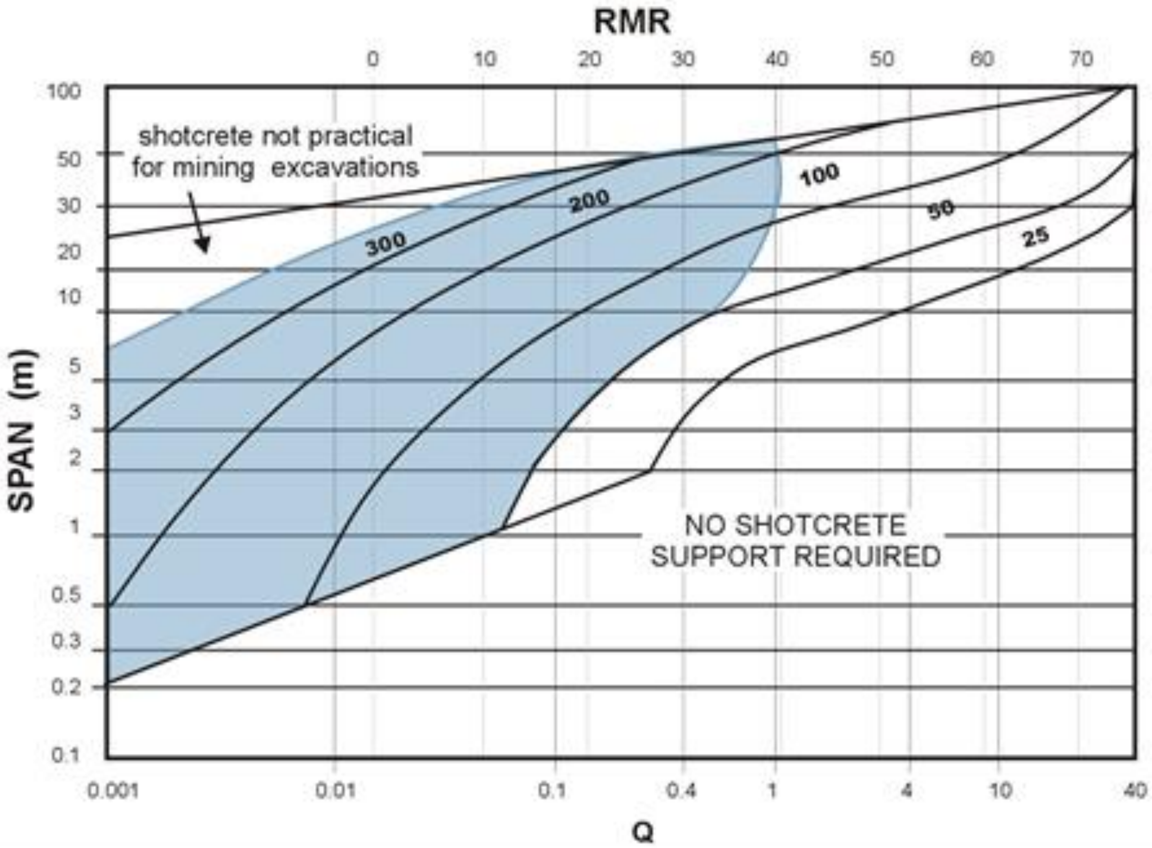


Figure 4-43 Mining approach for shotcrete (based on Q and RMR)

Table 4-30 Support requirements as per mining approach

	Rock Bolts	Shotcrete
Declines (5.5m x 5.8m)	Length 1.8m No Support to 1 bolt every 3m²	0 mm
Drives (4.5m x 4.5m)	Length 1.5m No Support to 1 bolt every 3m²	0 mm

4.14.3 Wedge Analysis

The software program Unwedge was used to analyse the wedge stability potential and determine ground support requirements. The structural sets determined from the structural assessment were adopted for use in the wedge analysis.

Methodology and Assumptions

In geotechnical domains, where it was interpreted that three or more structural sets exist, an Unwedge analysis was completed.

The assumptions that were used in the Unwedge analysis include:

- defect sets determined from the structural assessment were able to exist locally in the rock mass
- rock density of 2.7 t/m^3
- defect lengths were assumed to be discontinuous
- groundwater pressure assumed to be zero
- defect strength assumed to be cohesionless (i.e. c = 0 KPa), with friction angle of 35^o
- development dimensions are 5.5 m (width) x 5.8 m (height)
- no regional stresses were taken into account
- shotcrete was not used during the wedge analysis.

The assumptions regarding the strength characteristics for ground support elements are defined in Table 4-31.

Table 4-31 Ground support strength characteristics-wedge analysis

Bolt Type	Split Sets (47mm)	Cable Bolt
Tensile Strength (tonnes)	17	25
Bond Strength (tonnes/m)	3.5	12
Plate Strength (tonnes)	10	10

Analysis and Summary of Wedge Potential

Eight structural sets were encountered in the footwall domain. A wedge analysis was performed for the three sets which would result in a realistic worst case wedge. The analysis assumed a development profile of 5.5 m (width) x 5.8 m (height), which is the largest potential development profile and a four way intersection.

The wedge potential will vary with different orientations of the proposed development; therefore, the wedge potential was assessed for each of the potential trends that the proposed development would take.

The wedge weight describes the maximum size wedge possible while the support pressure indicates the tensile capacity of the bolts per square metre, required to retain the wedge to a Factor of Safety (FoS) of 1.5.

The largest wedge formed on the south portal decline was 225 tonnes; continuity of structures is required to form wedges to this size otherwise smaller wedges may form which will be readily controlled by surface support or by dressing the excavation.

A ground support design comprising a pattern of 2.4 m length Split Sets (47 mm diameter) on a 1.4 m by 2.0 m spacing was applied to the development axis trends.

After ground support is applied, wedges below a Factor of Safety of 1.5 remain. This typically results from numerous wedges identified, existing between bolt locations (i.e. smaller wedge sizes). Surface support in the form of welded mesh or shotcrete offers a minimum of 2 tonnes support between bolts for a 1.7m by 2.0m bolt spacing. Therefore, surface support would control the identified smaller wedges in Figures J3 to J5 where the required support pressure does not meet the requirements.

The smaller wedges will most likely be dislodged during the blasting and/or scaling of the excavation.

4.14.4 Support Recommendations

Central and Eastern Declines

Detailed analysis of the different geotechnical areas indicated similar support requirements for each area. The general designs for the declines indicated that a 1.8 m long split set on a basic 1.5 m x 2.0 m pattern is sufficient for all the areas. However, it is an industry standard to increase the length of the split set by 30%, due to the critical relationship between split set and hole diameter, and it is recommended that 2.4 m split sets be used to minimise wedge fallouts.

No shotcrete is required for the general expected conditions in the declines.

The support recommendations are based on the two ground support design approaches, as well as practical experiences.

The following ground support design (Figure 4-44) is planned for the first 100 m of each decline, after which the pattern will be reassessed based on the exposed rock conditions:

- A 25 mm thick layer of shotcrete will be applied soon after blasting each round due to the weathering nature of the shale.
- Split sets (2.4 m x 47 mm) installed on a 1.4 m x 2.0 m rectangular pattern.

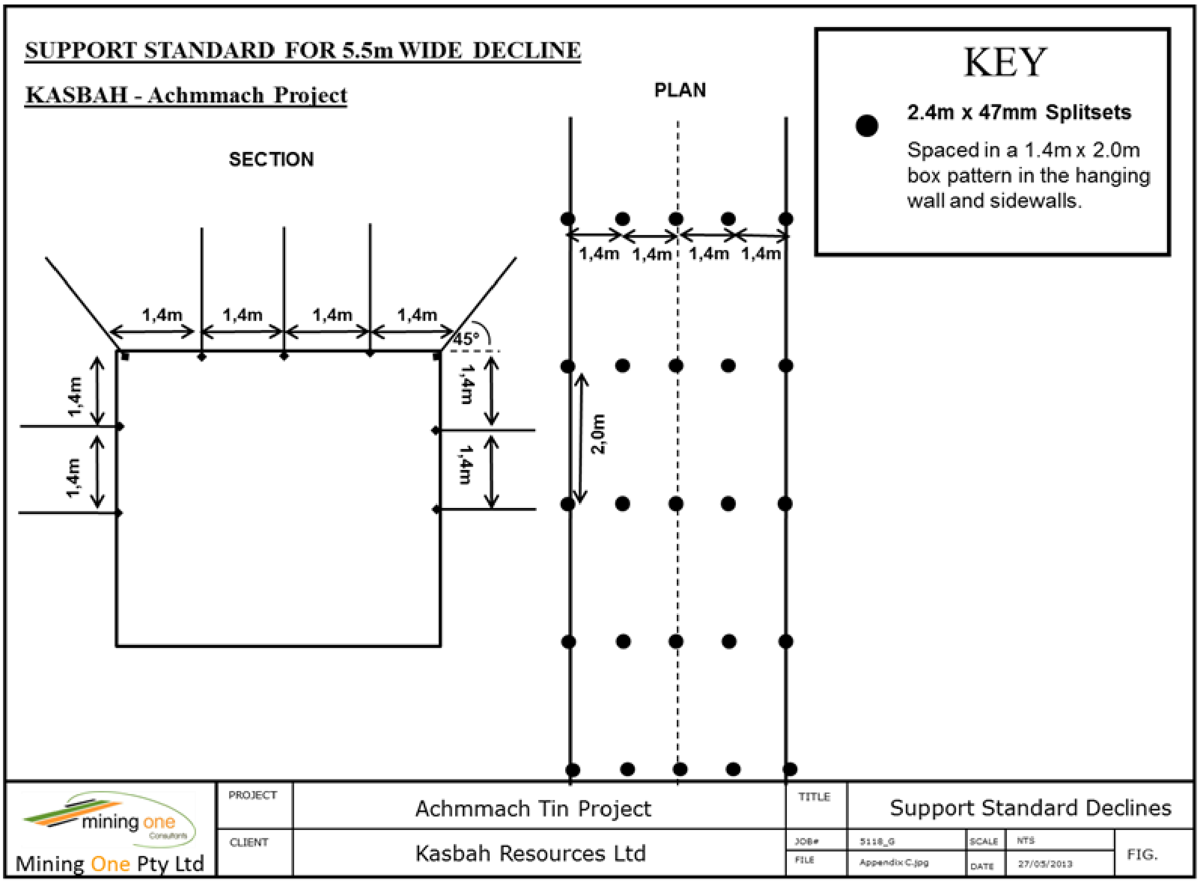


Figure 4-44 Decline support

The potential to reduce the support after the first 100 m of development to the secondary support type of split sets: 2.4 m x 47 mm on a 1.4 m x 2.0 m box pattern may exist. No shotcrete or mechanical anchors will be required.

Additional support will be required within localised areas of poor ground conditions which could be related to faults or shear zones. The exact location of these faults and shears are currently not known but will be identified by geologists and geotechnical engineers during the drilling and development phases of the declines and drives. Site specific support for these faults and shear zones will be designed by a geotechnical engineer.

Drives

Support designs for the drives followed similar procedures as recommended for the declines.

The following support design is recommended for the drives (Figure 4-45):

- no shotcrete
- split sets (2.4m x 47mm) installed on a 1.5 m x 2.0 m rectangular pattern

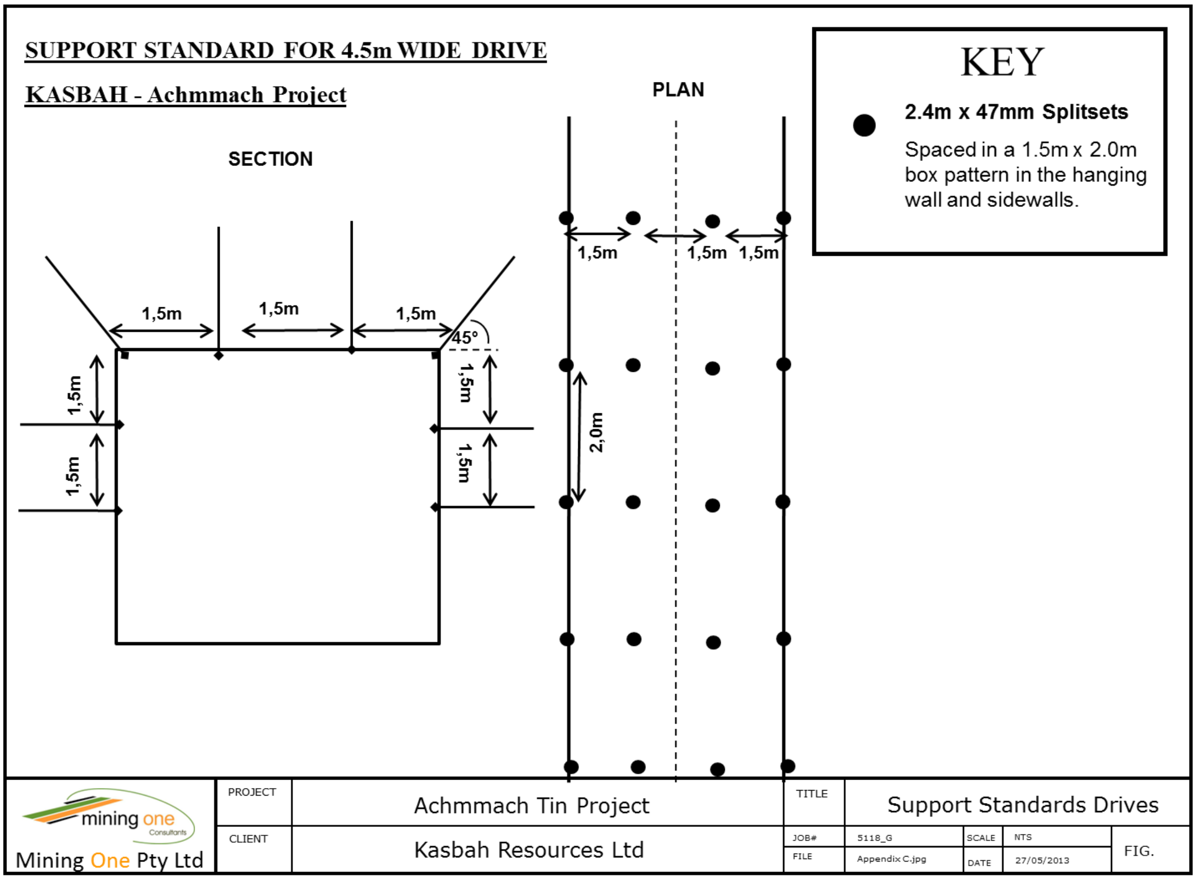


Figure 4-45 Ore drive support

Additional support will be required within localised areas of poor ground conditions. A geotechnical engineer will design support specific to site conditions.

4.15 Stability of Raise Bored Ventilation Shafts

A method of assessing the geotechnical risk for large diameter raise bores or shafts was developed by McCracken and Stacey (1989). It was developed by contractors in South Africa and Australia and is widely used in the mining industry. The method is also applicable to the stability evaluation of conventionally excavated shafts. The method is based on the Q System, with adjustments applied to take account of orientation of joints and weathering.

These adjustments are given in Table 4-32 and Table 4-33 below.

Table 4-32 Adjustment for joint orientations

Face Orientation Adjustment				Wall Orientation Adjustment			
No of flat dipping (0o-30o) major joint sets	1	2	3	No of steep dipping (600-900) major joint sets	1	2	3
Adjustment	0.85	0.75	0.60	Adjustment	0.85	0.75	0.60

A wall adjustment different from that indicated above for the general Q System is applied:

Qwall = 2.5 x Q when Q > 1

Qwall = Qwhen Q < 1.

The average rock mass ratings of the shafts were risk assessed in order to determine the Q and QR values and associated probability of failure percentages (Table 4-33 and Figure 4-46).

Table 4-33 Rock quality (QR) and reliability for raise bore shafts (4.5m)

	Thickness	Ave Q	Wall factor	Joint factor (3 sets)	Weathering factor	QR rock quality	Probability of failure	Reliability 4.5m
Saprock	5m	.5	2.5	0.6	0.75	0.6	0.25	75%
Hard Rock 25%	+300	2.2	2.5	0.6	0.9	3.0	0.05	95%
Hard Rock 50% (Mean)	+300	5.6	2.5	0.6	0.9	7.6	0.02	98%
Hard Rock 75%	+300	9.2	2.5	0.6	0.9	12.4	0.01	99%

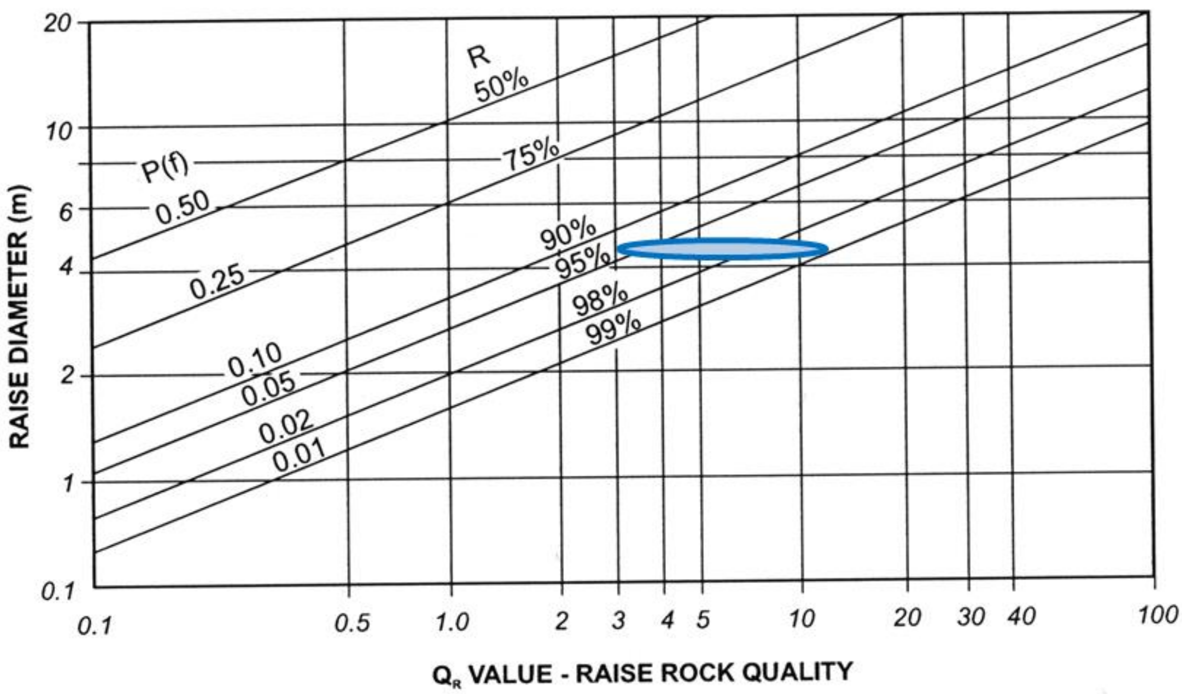


Figure 4-46 Risk ratings for the raise bore shafts (Diameter 4.5m)

For the hard rock a reliability (R) of 95%-99% and a probability of failure P(f) of 0.05 – 0.01 is considered acceptable. The saprock will be cleared during the pre-sink down to contact of the hard rock.

Geotechnical holes will be drilled at the proposed ventilation shaft positions and geotechnical assessment completed prior to raise boring, to determine the site-specific ground conditions, risk zones, achievable diameters, ground support requirements etc.

4.16 Conclusions

Notwithstanding some of the problems identified in the development of the geomechanical models, the models and analysis are comprehensive and demonstrate that the ground conditions are anticipated to be manageable with typical ground support and stope support techniques.

Further laboratory tests will be conducted to compile a complete range of UCS values against the bedding and foliation intersection angles. This would serve to validate the current analysis.

During initial stages of mining the geomechanical properties of the mine waste as to its suitability to be used as fill material. This will serve to validate the current analysis and provide important data for further optimisation.

The mining methods proposed were found to be geotechnically sound and technically viable.

5 MINING

5.1 Introduction

The geological and geotechnical analysis of the Achmmach tin deposit indicates the following:

- The lodes form a 300 metre wide array across strike with individual lode structures ranging in width from one metre to 30 metres as shown in Figure 5-1 below.
- Tin mineralisation occurs primarily as breccia infill and quartz-cassiterite veins.
- The ground conditions are generally good and will not significantly constrain the development and stoping design.

These physical characteristics and the work completed to date support the concept for a mechanised underground mining operation.

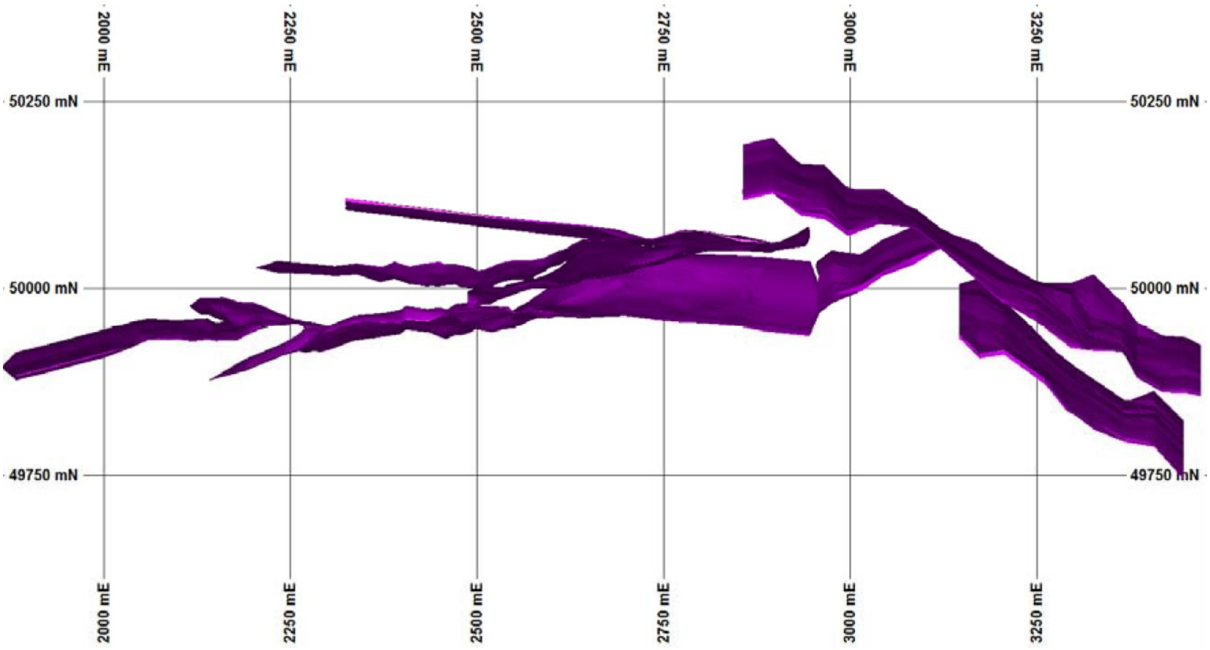


Figure 5-1 Plan view of vertical feeder mineralisation

Previous mining studies on the Achmmach deposit include a definitive feasibility study (DFS) in 2014, an enhanced DFS (EDFS) in 2015, and a low cost small start option (SSO) in 2016. This work was reviewed by AMC Consultants in 2017 to identify opportunities to further optimise and de-risk the project. Following on from this review, Entech (mining consultant) was engaged to undertake several design and scheduling changes (Appendix 5A) including:

- A review of the mine cut-off grade strategy.
- A review of the geotechnical design parameters across the mine design to ensure these are consistently applied.
- A review the mine cemented rock fill strategy and approach including key mining productivities.
- Incorporation of grade control diamond drilling into the schedule as a constraint and a reported physical item.

These revisions and improvements resulted in this definitive feasibility study for the Achmmach project. The study mine plan includes 7.0 Mt of ore grading 0.82% Sn for a total of 57,600 t of tin metal over a mine life of 126 months (10.6 years).

A summary of the mine plan physicals is presented in Table 5-1 and the design is shown in Figure 5-2.

Table 5-1 Summary of mine plan physicals

Parameter	Achmmach Mine Plan
Mine life	10 years
Production rate (tpa)	750 ktpa
Mining method	Bottom-up CRF and top-down open stoping
Stope shape basis	Automated optimisation software followed by manually review
Stope dilution (m)	0.5 m on each contact
Average stope recovery (%)	93.7%
Capital lateral development (m)	20,219
Operating lateral development (m)	24,093
Total development (m)	44,312
Development ore (t)	1,076,543
Development ore grade (% Sn)	0.72%
Development ore metal (t Sn metal)	7,736
Stope ore tonnes (t)	5,936,800
Stope ore grade (% Sn)	0.84%
Stope ore metal (t Sn metal)	49,910
Total ore tonnes (t)	7,013,344
Total ore grade (% Sn)	0.82%
Total ore metal (Sn)	57,645
Production drilling (m)	693,414
Diamond drilling (m)	75,925
Cemented rock fill placed (t)	1,018,698
Rock fill placed (t)	875,890
Total haulage (tkm)	14,182,555

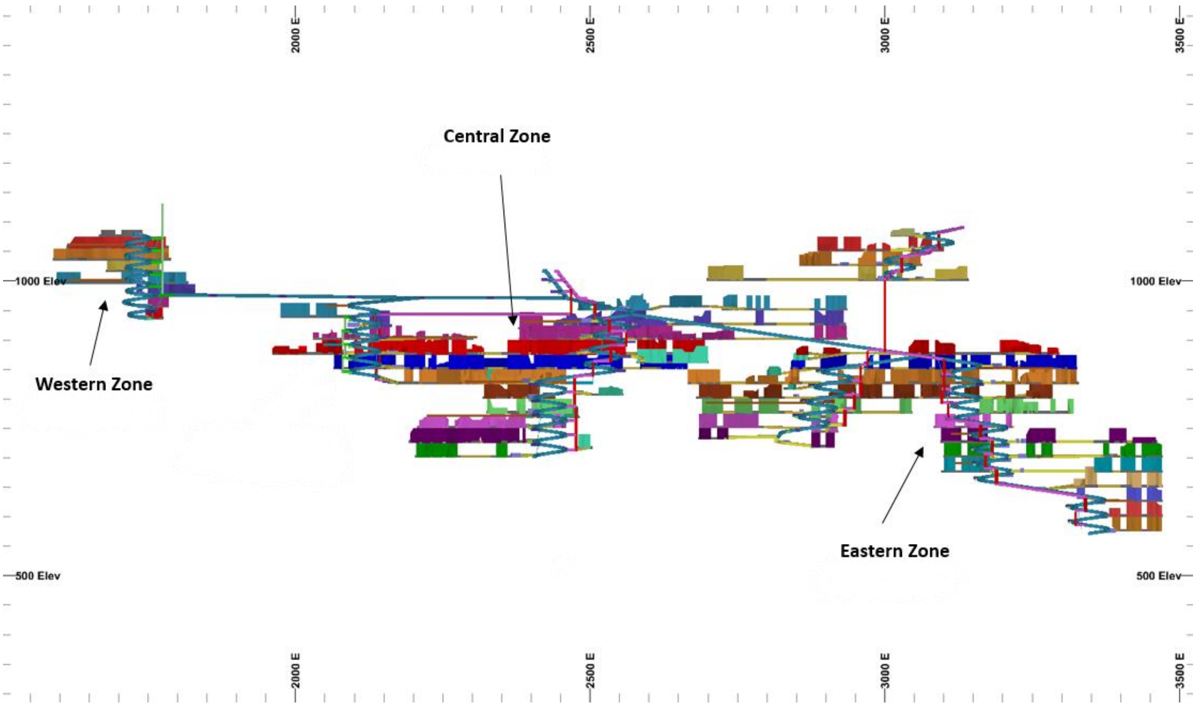


Figure 5-2 Achmmach mine plan long section

5.2 Method Selection and Preliminary Planning

The University of British Columbia has created a mining method selection tool which considers the unique characteristics of the mineralisation including its siting, geometry, grade distribution and the ground conditions of the mineralisation and surrounding rock mass to recommend suitable mining methods. It provides an objective, albeit simple, method for initial mine planning.

Table 5-2 UBC mining method selection

Characteristic	
Geometry and Grade Distribution	
General Shape:	Irregular
Ore Thickness:	Intermediate 10-30m
Ore Plunge:	Steep >55
Grade Distribution:	Erratic
Depth:	Intermediate 100-600m
Rock Mass Rating	
Ore Zone:	Strong (60-80)
Hanging Wall:	Strong (60-80)
Footwall:	Strong (60-80)
Rock Substance Strength (unconfined compressive strength / principal stress)	
Ore Zone:	Medium (10-15)
Hanging Wall:	Medium (10-15)
Footwall:	Medium (10-15)

The selection process allocates a score to each mining method according to how suitable that method is for each of the characteristics. The result for Achmmach was that sublevel stoping is the preferred method with a score of 37 and cut and fill stoping is next preferred with a score of 36.

This result was used to develop a mining concept on which to estimate mine production rates and mine operating costs which can feed into initial cut-off grade assumptions.

5.3 Cut-off Grade

The optimised cut-off grade was estimated by completing basic comparative mine schedules for mine inventories created by applying a range of cut-off grades between 0.5% Sn to 0.65% Sn. This exercise indicated maximum NPV at a cut-off grade of 0.55% Sn.

Analysis of previous financial modelling indicates that the updated mine and processing plan provides a lower COG of 0.37% Sn. Future iterations could potentially provide increased resource to reserve conversion by applying this COG. Ideally COG is altered over time to maximise NPV, with initially higher then declining COG values. Due to the uncertainty of commodity price forecasts this exercise is generally not completed at the DFS stage or would in any event be revised in the course of mining.

5.4 Mining Method

The study work to date has determined that the preferred mining method is conventional mechanized longhole stoping with design variations used for backfilling or pillars. As the geometry and thickness of the mining shapes vary throughout the different lodes, a combination of bottom-up cemented rock fill (CRF) and top-down open stoping methods is planned to provide the necessary ground support for the varying conditions.

The proposed mine design employs CRF in areas of higher grade and greater ore width to minimise metal loss to pillars, with the lower cost open stoping method used in the areas developed later in the life of mine schedule. For scheduling purposes, the Western Zone had both methods applied, with bottom-up CRF above 1015 mRL and top-down open stoping below this point.

5.4.1 Central Zone Mining Method

For the generally thick Central Zone (up to 20 m wide), a bottom-up mining sequence utilizing CRF is planned to be used. A top-down method using pillars for stability instead of backfill was also analysed but this design significantly reduced recovered ore tonnes due to the required widths of the pillars. Stoping will be carried out retreating from the ore drive extremities back to a central access.

The planned method allows multiple concurrent working areas (or panels) to be mined off the same decline as stoping in each panel advances upwards underneath the bottom cemented fill stopes in the panel above. This allows stoping to commence after development of three to four levels, rather than requiring development to the bottom of the mine before commencement of stoping. Figure 5-3 illustrates the panels designed in the Central Zone.

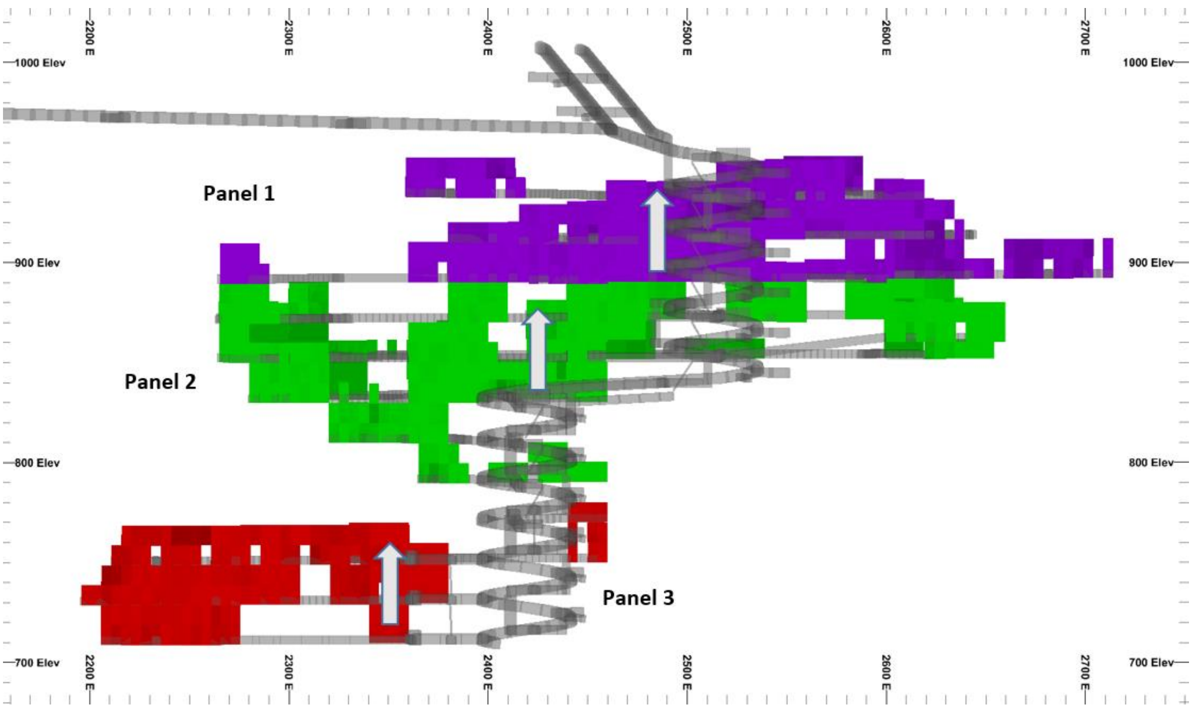


Figure 5-3 Central zone schedule showing concurrently mined panels (long-section looking north) (coloured by panel)

5.4.2 Eastern Zone Mining Method

The Eastern Zone area was designed to be mined top-down using pillars for stability as shown in Figure 5-4. The pillar factors have been extrapolated across the entire Eastern Zone area from the Eastern Zone Upper. This is considered reasonable as the ground conditions will improve slightly with depth to offset any increase in ground stress.

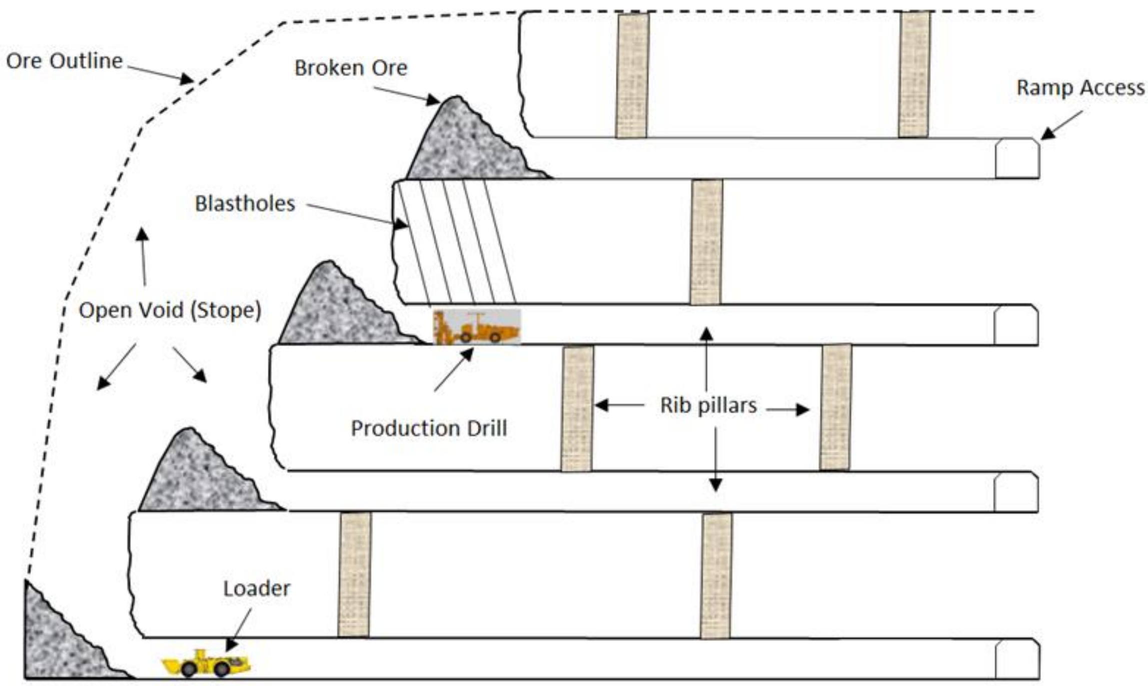


Figure 5-4 Top-down open stoping method

5.4.3 Western Zone Mining Method

The Western Zone mining method adopted both the CRF bottom-up method (for levels above the access) and the longhole top-down method (for levels below the access) as described above.

5.4.4 Method Summary

Table 5-3 summarises the methods selected for each mining area and Figure 5-5 illustrates the application of fill strategy throughout the mine.

Table 5-3 DFS mining methods

Mining Zone	Mining Method
Central Zone	Bottom-up CRF
Central Western Zone	Top-Down Open Stoping with Pillars
Western Zone (above 1015mRL)	Bottom-Up CRF & Top-Down Open Stoping with Pillars
Eastern Zone	Top-Down Open Stoping with Pillars

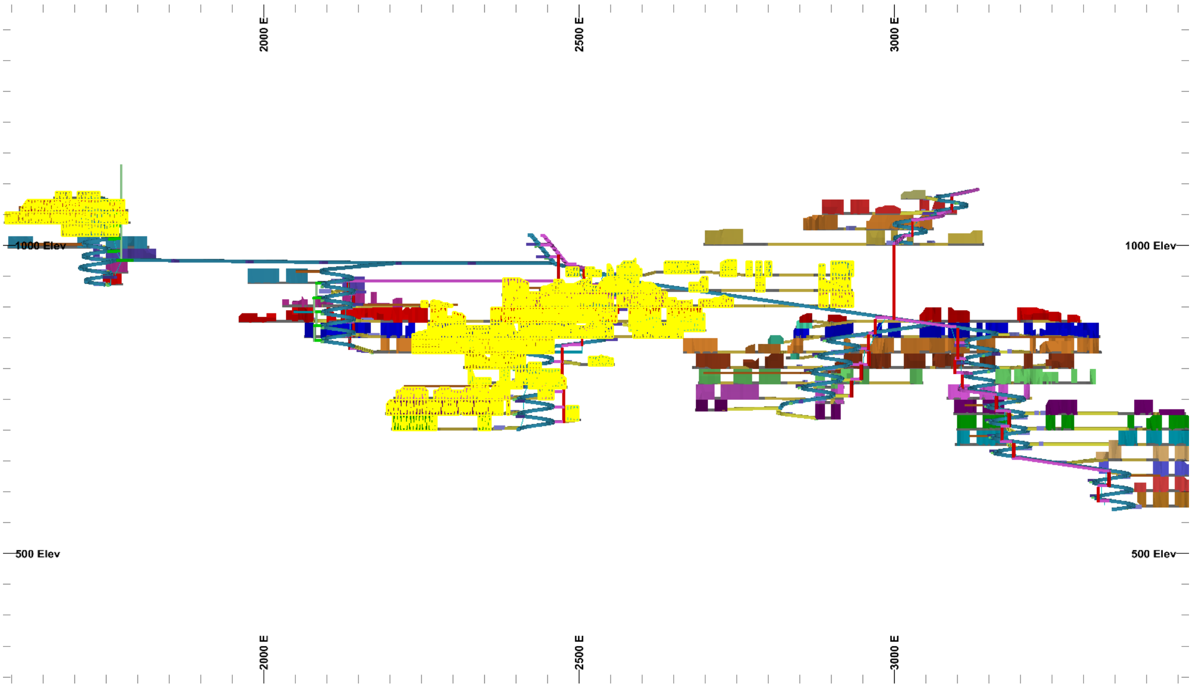


Figure 5-5 Mining methods by zone, CRF highlighted yellow otherwise top down with pillars

5.5 Geotechnical Design

5.5.1 Development Support Design

The derivation of the development ground support regimes is discussed in Section 4.14. These regimes were used for determining jumbo productivity and for cost estimation. The regimes are typical for good ground conditions and are presented in Figure 5-6 and Figure 5-7.

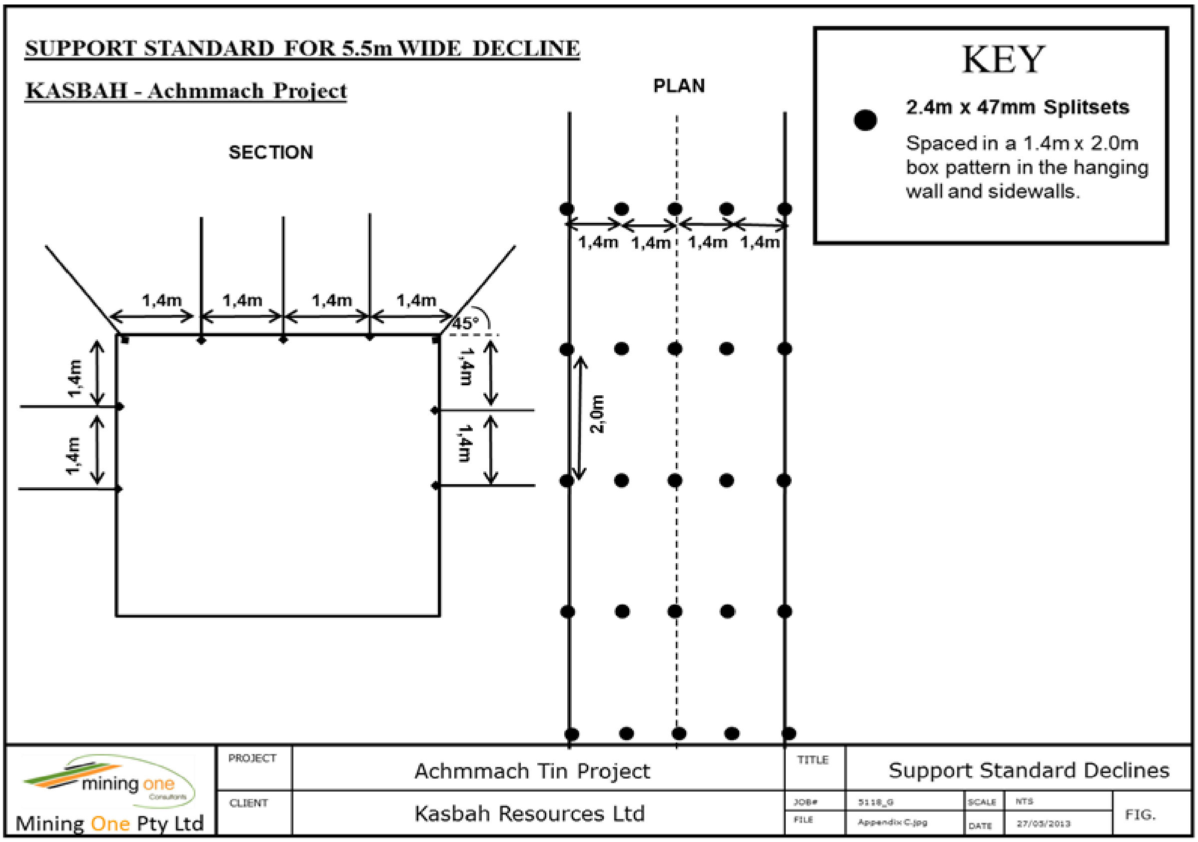


Figure 5-6 Decline support preliminary design

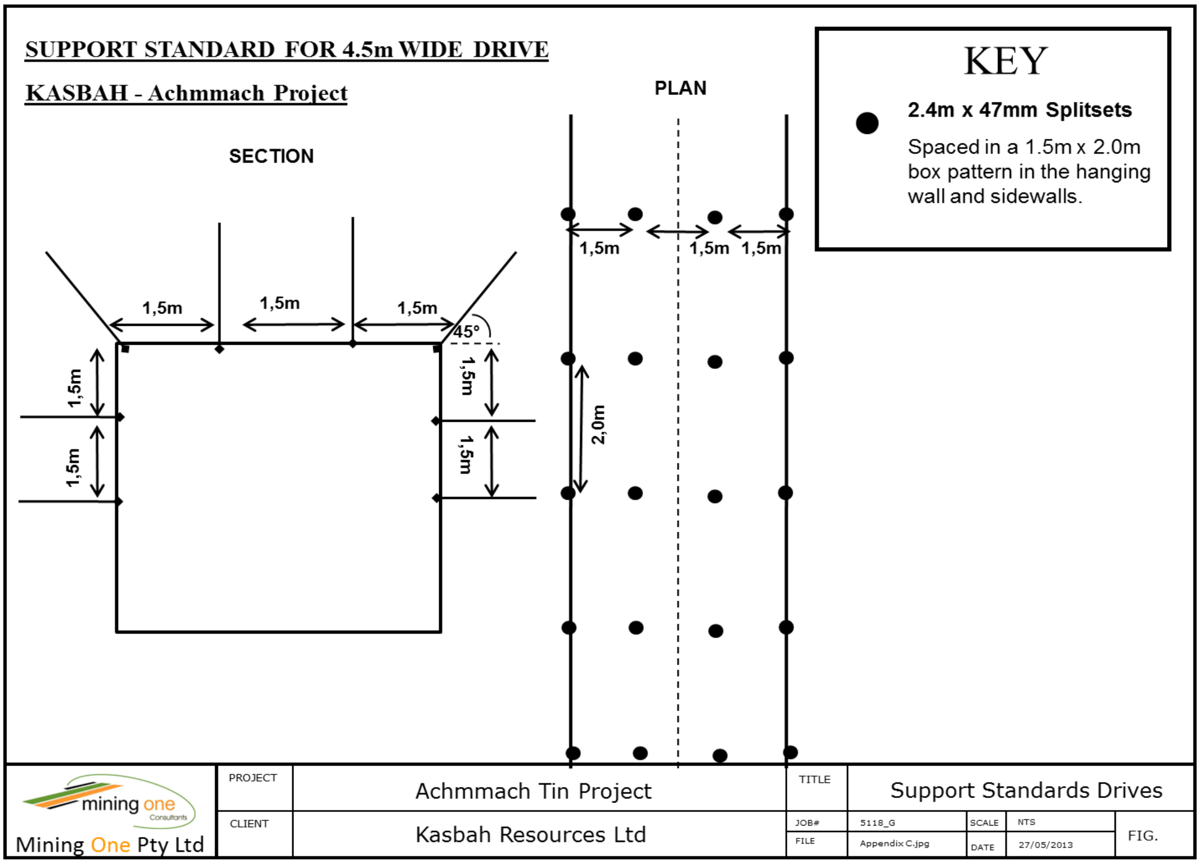


Figure 5-7 Ore drive support preliminary design.

To account for the expected amount of additional support over and above the base plan, 50 mm thick shotcrete has been assumed to be required over 5% of the total development length of the mine. All intersections have been assumed to require installation of eight x 6 m cable bolts.

5.5.2 Stope Stability

Maximum stope span recommendations are based on the stability graph method. These are graphs for the Meknès and Eastern Zone areas, presented Section 4.10.1 and summarized in Table 5-4, Table 5-5 and Table 5-6.

Table 5-4 Back span estimates for the initial stope design purposes.

Width	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
9m	+100m	-	+100m	-
25m	-	30m	-	+100m

Table 5-5 Stope height estimates for the initial stope design purposes inclination 55°

Height	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
30m	60m	60m	+100m	+100m
60m	30m	30m	50m	50m
90m	25m	25m	40m	38m

Table 5-6 Stope height estimates for the initial stope design purposes inclination 90°

Height	Unsupported		Supported	
	Eastern	Meknès	Eastern	Meknès
25m	90m	90m	+100m	+100m
50m	35m	35m	60m	60m
75m	27m	27m	42m	42m

Pillar width designs were generated by determining the rock pillar strengths and the stresses acting on the pillars with an appropriate factor of safety applied. The resulting pillar width recommendations are summarized in Table 5-7 and Table 5-8.

Table 5-7 Recommended open stope pillar sizes for the upper Eastern Zone.

Width	Sill		Rib	
	East	Meknès	East	Meknès
3m	3m	N/A	3m	N/A
6m	6m	N/A	6m	N/A
9m	9m	N/A	9m	N/A
12m	12m	N/A	12m	N/A

Table 5-8 Recommended open stope pillar sizes for the lower Eastern Zone

Width	Sill		Rib	
	East	Meknès	East	Meknès
3m	6m	N/A	7m	N/A
6m	9m	N/A	11m	N/A
9m	12m	N/A	14m	N/A
12m	15m	N/A	18m	N/A

5.5.3 Boxcut Stability

The box cut was designed as in accordance to the geotechnical parameters outlined by the geotechnical analysis Section 4.13 and summarized in Table 5-9.

Table 5-9 Geotechnical boxcut design parameters

	Central Box Cut	East Box Cut
Soil	4m bench at 45° 4m berm	4m bench at 45° 4m berm
Saprolite	8m bench at 60° 4m berm	8m bench at 60° 4m berm
Fresh Rock	10m bench at 90°	10m bench at 90°
Portal	5.5m * 5.8m decline 5m solid beam ± 1 – 2m from the box cut sidewall	5.5m * 5.8m decline 5m solid beam ± 1 – 2m from the box cut sidewall

5.6 Stope Optimisation and Design

The stope shapes used in this iteration of the mine plan were based on the 0.55% Sn COG from the 2015 EDFS. The Datamine Mineable Shape Optimiser® (MSO) software was used to generate the initial shapes using the parameters detailed in Table 5-10. All stopes were manually reviewed and modified where necessary to ensure practical mineability, considering:

- Local orebody geometry and grade distributions;
- Practical mining requirements for long hole drilling and maximum recovery from stope bogging;
- Minimisation of designed dilution; and
- Geotechnical stability parameters as discussed in Section 4.10.

Table 5-10 MSO parameters

Parameter	Meknès Assumption	WZ Assumption
Minimum stope width (m)	3.0	3.0
Maximum stope width (m)	20.0	20.0
Minimum dip (deg.)	40.0	40.0
Stope height (m)	25.0	20.0
Section strike length (m)	5.0	5.0
Min. waste pillar width between stopes (m)	5.0	5.0

Total Dilution (m)*	0.5	0.5
Cut-off grade (% Sn)	0.55	0.55

*The dilution applied in the optimisation phase was increased to allow for a total 1 m dilution skin in the final mine plan.

The final stope shapes are depicted in Figure 5-8 and Figure 5-9.

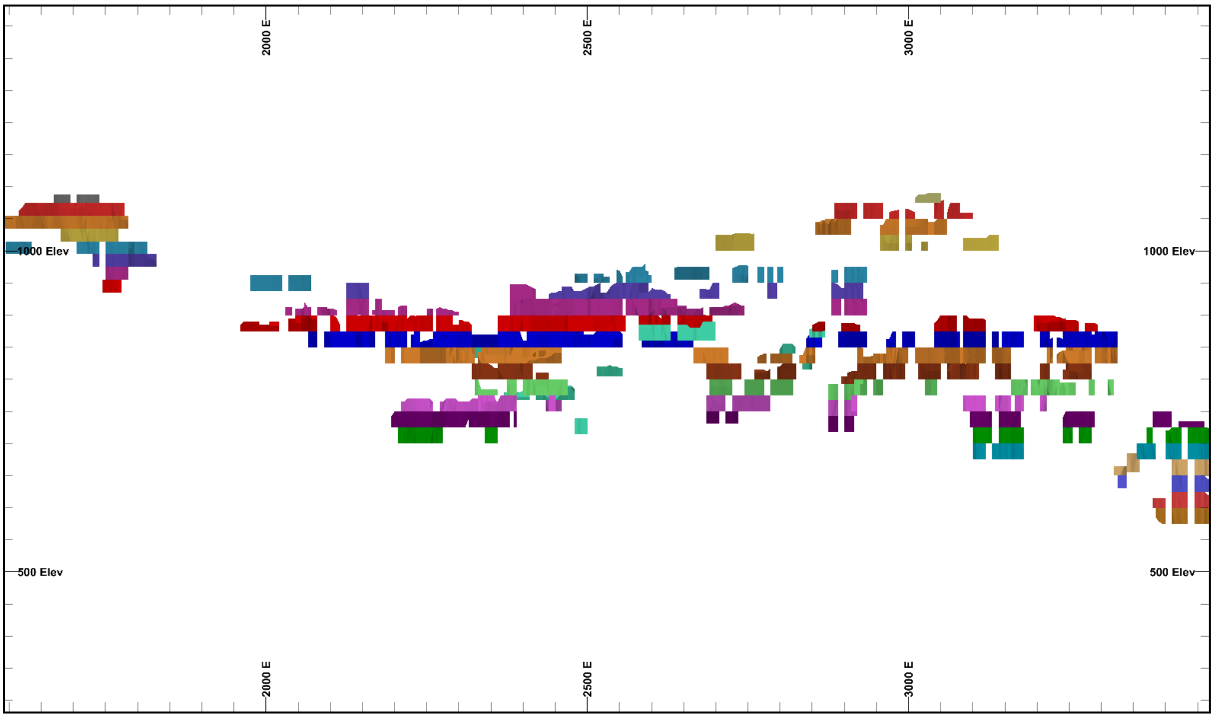


Figure 5-8 Long-section stope design (looking north coloured by RL)

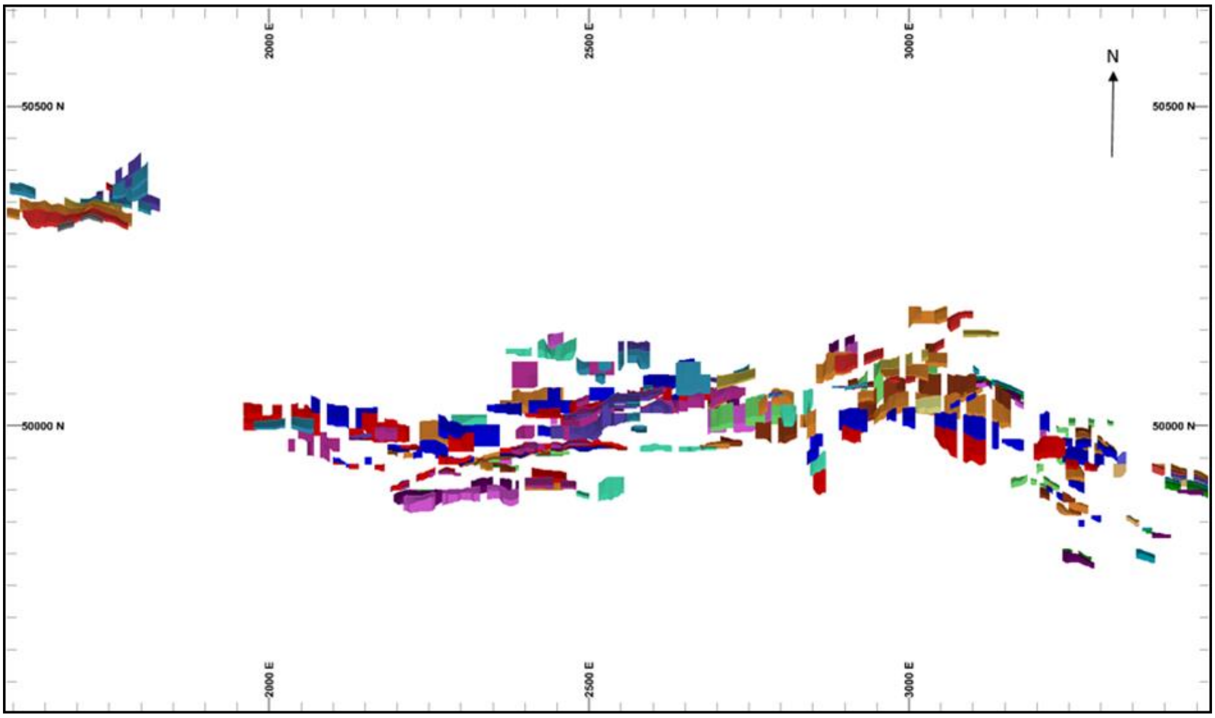


Figure 5-9 Plan view stope design (coloured by RL)

5.7 Development Design

The Achmmach Mine is accessed via two locations from the surface,

- The Central Portal Box cut @ 1015 mRL and;
- The Eastern Portal Box cut @1085 mRL

Key features of the final design are:

- At each boxcut location twin portals and declines are designed, providing ventilation and escapeway drives parallel to the decline. This eliminates the requirement for raisebored ventilation raises to surface in the Central and Eastern Zones early in the mine plan;
- Three internal declines servicing the Western Lodes, Eastern Lodes and Central Lodes.
- Independent escapeway drives are developed laterally as a part of each level development and vertically through 1.1 m raises to provide a second means of egress from the production levels once stoping commences.
- The stand-off distance from the decline to the stopes is greater than 25 m, based on geotechnical analysis discussed in Section 4.12.2.
- Diamond drill drives are designed to provide appropriate drilling platforms for grade control drilling programs.

The final development design is shown in Figure 5-10.

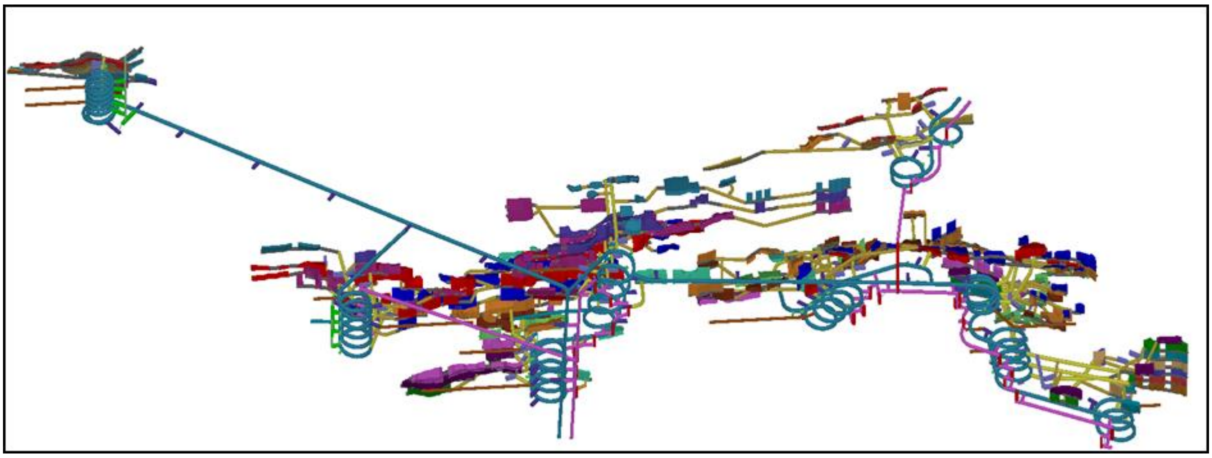


Figure 5-10 Isometric view of development and stope design. Development coloured by type and stopes coloured by RL

5.7.1 Capital Lateral Development

Stockpiles for each level will be located along the level crosscut to provide storage for broken material and a truck loading point. Crosscuts are nominally 30 m long with the same profile as the decline to allow truck access to these level stockpiles. Sumps have been designed on each access.

Decline stockpiles will be developed with every second one being broken through into the fresh air raise for primary ventilation flow.

Level stockpiles will be mined to approximately 7 m high to allow back-tipping 50 t trucks to dump waste for stope filling. A typical level design from the decline is shown in Figure 5-11 and in section view in Figure 5-12.

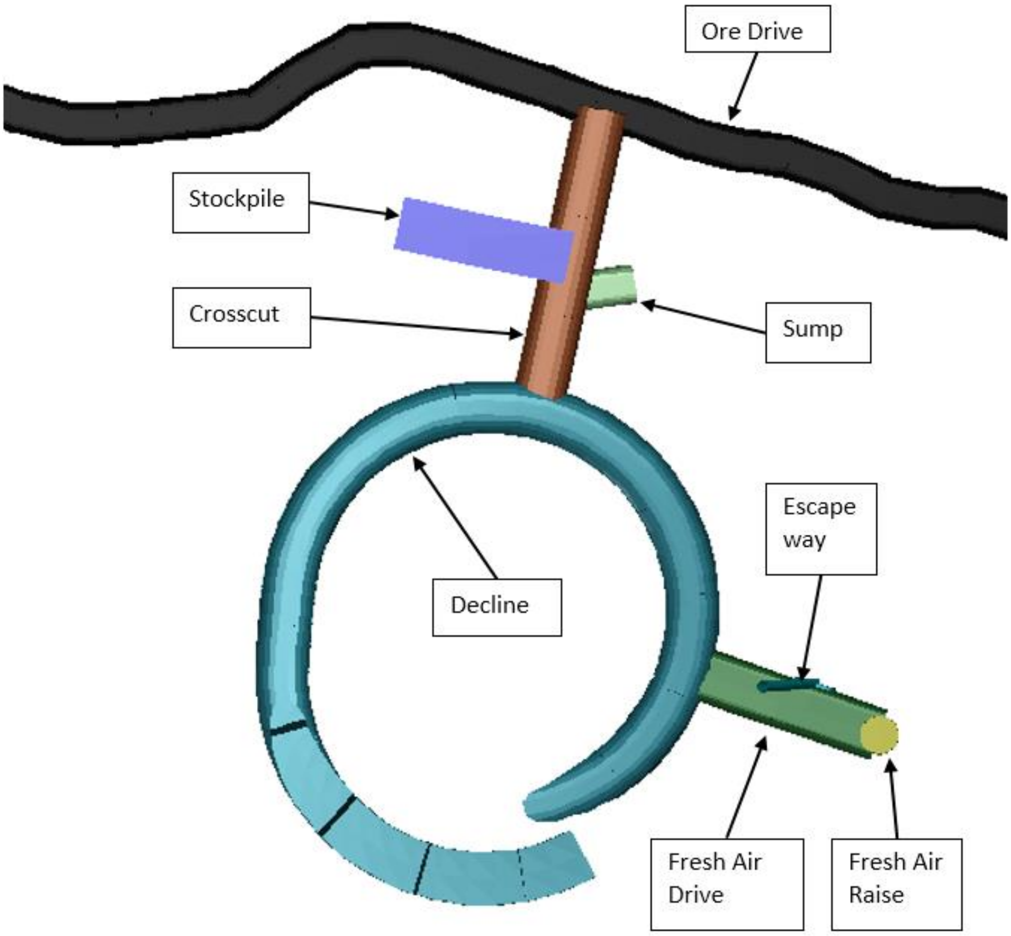


Figure 5-11 Typical level layout-plan

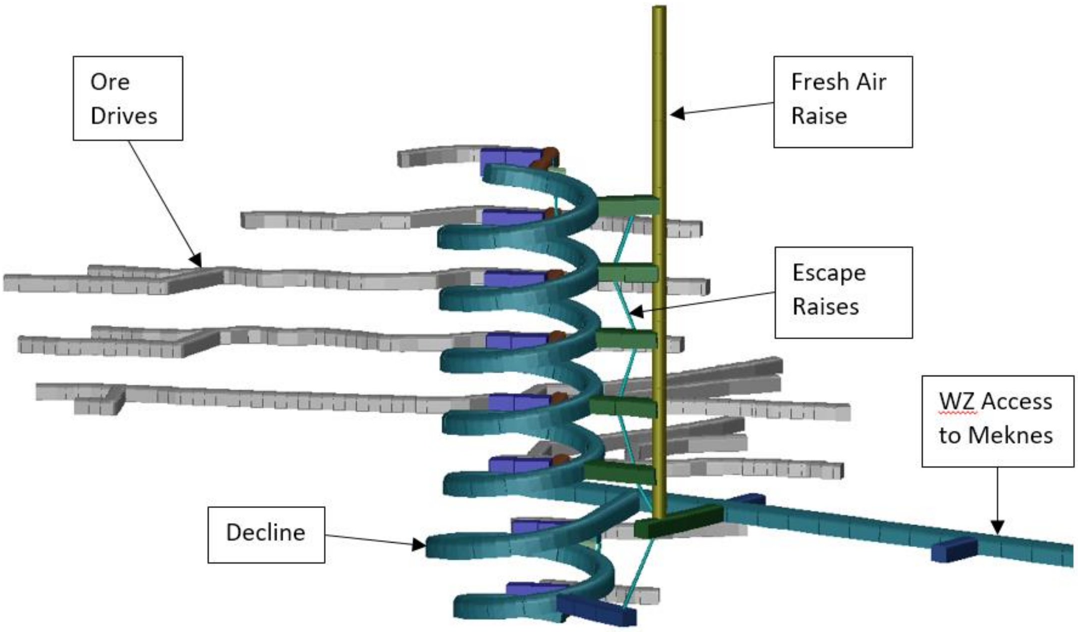


Figure 5-12 Typical mine layout-section

5.7.2 Ore Drives

Ore drives will be developed at a nominal 4.5 mW x 4.5 mH. The drive width is based on the manufacturer recommendations for the proposed Caterpillar R2900 production loaders and the drive height to accommodate the proposed drills and to provide sufficient clearance for secondary ventilation ducting.

5.7.3 Vertical Development

Slot raising for opening new stopes will be undertaken using a production drill rig. Slot raises have not been designed as this is part of the production cycle, however estimates for drilling the slot raises have been included in the schedule.

Escapeways have been designed at 1.1 m diameter to be developed using airleg or longhole techniques to accommodate a prefabricated ladder system.

The total development for the mine plan is presented in Table 5-11.

Table 5-11 Mine plan development physicals

Development Type	Dimensions	Approximate design metres
Decline	5.5 mW x 5.8 mH	9,300
Stockpiles	5.5 mW x 5.0 mH	2,300
Level accesses	5.5 mW x 5.5 mH	4,800
Operating waste drives	4.5 mW x 4.5 mH	4,600
Sumps	4.5 mW x 4.5 mH	400
Return air drives	5.0 mW x 5.0 mH	3,100
Escapeway drives	4.5 mW x 4.5 mH	400
Diamond drill development	4.5 mW x 4.5 mH	1,300
Total waste development		**26,200**
Ore drives	4.5 mW x 4.5 mH	11,500
Low grade ore drives	4.5 mW x 4.5 mH	6,700
Total ore development		**18,200**
Total lateral development		**44,400**
Raisebored ventilation raises	4.5 m diameter	400
Longhole ventilation raises	4.5m x 4.5 m	800
Escapeway raises	1.1 m diameter	1,000

5.8 Portal Area

The surface box cut footprint for the portal area includes the following aspects;

- Sufficient ore capacity for the mine ROM; and
- Sufficient capacity to temporarily stockpile waste for subsequent CAF requirements.

The plan layout in Figure 5-13 shows the box cuts in relation to the waste dump facilities and the ore ROM Pad at the Mill.

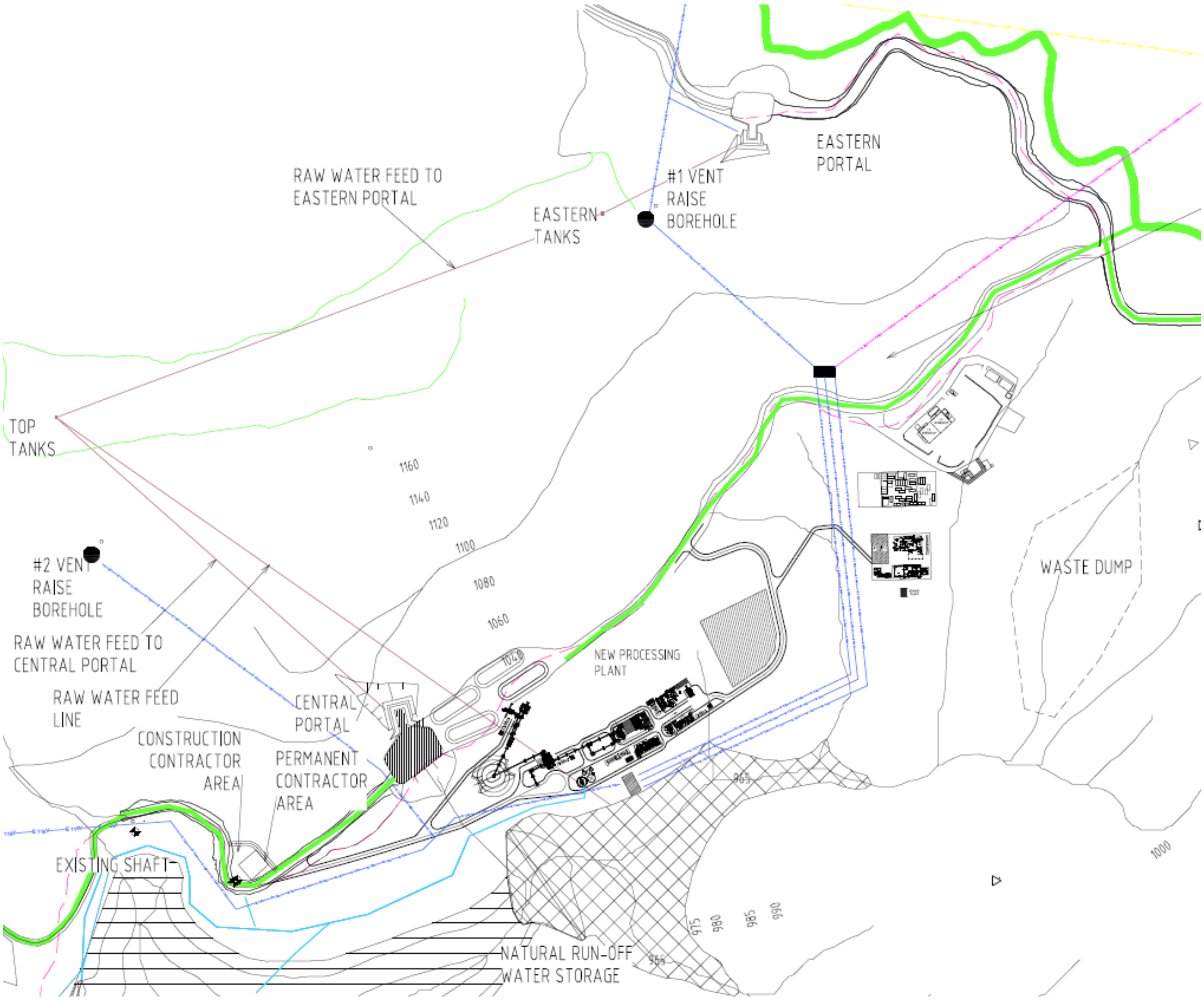


Figure 5-13 Portal layout in relation to key surface infrastructure

Isometric and plan views of the Central boxcut and apron are illustrated in Figure 5-14 and Figure 5-15.

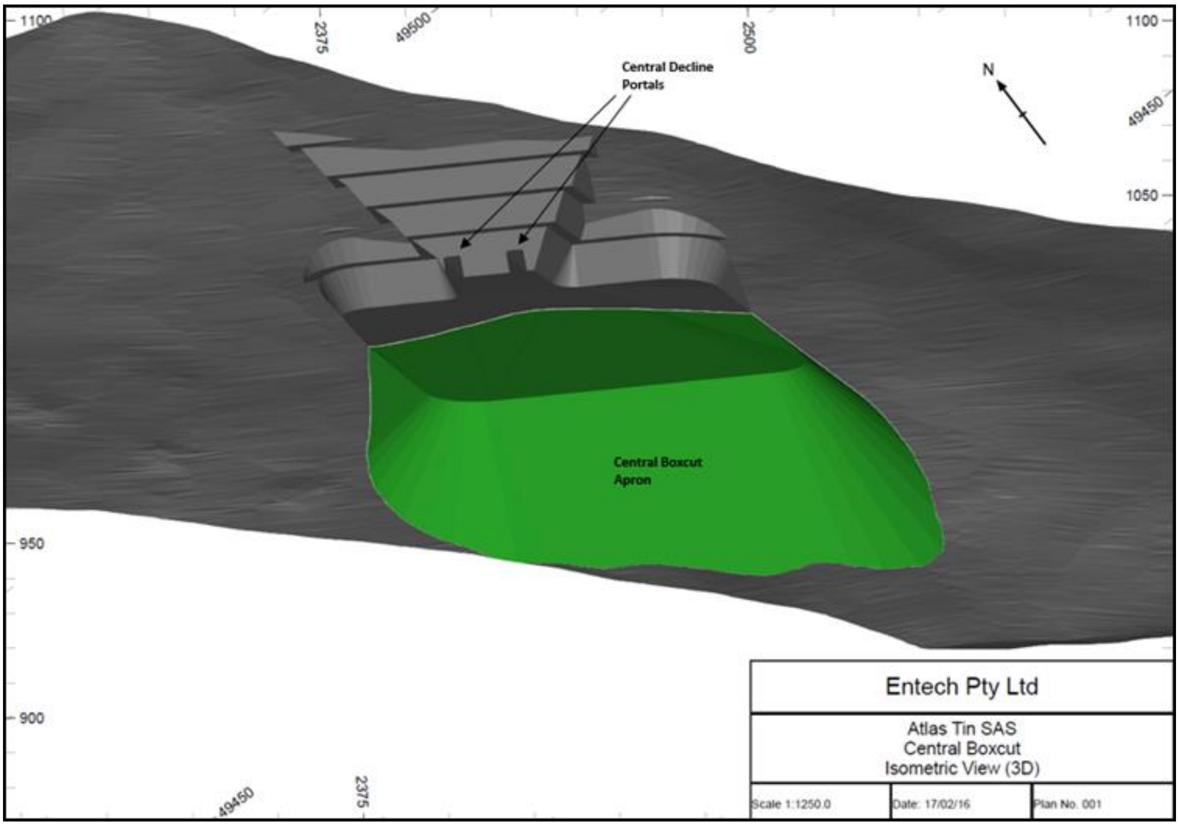


Figure 5-14 Isometric view of central boxcut and apron (looking NE)

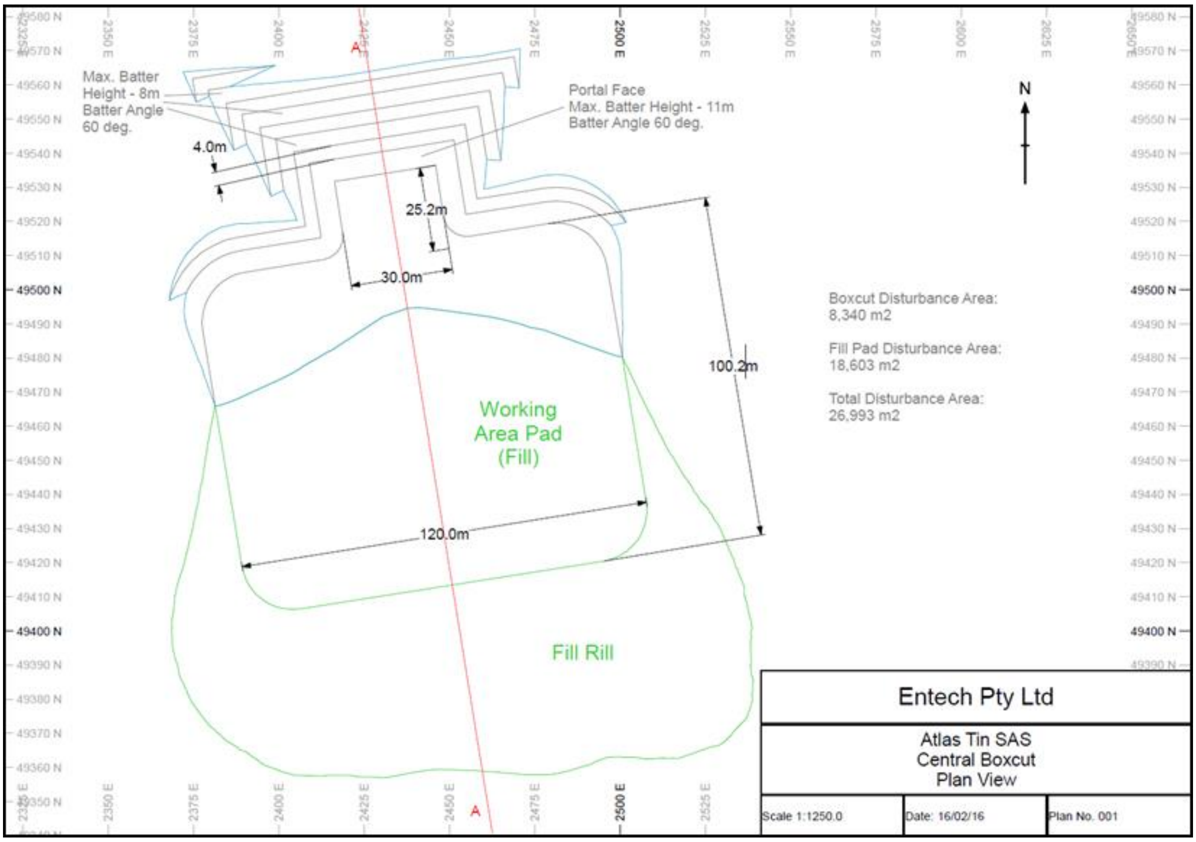


Figure 5-15 Plan view of central boxcut and apron

Isometric and plan views of the Eastern boxcut and portal are shown in Figure 5-16 and Figure 5-17.

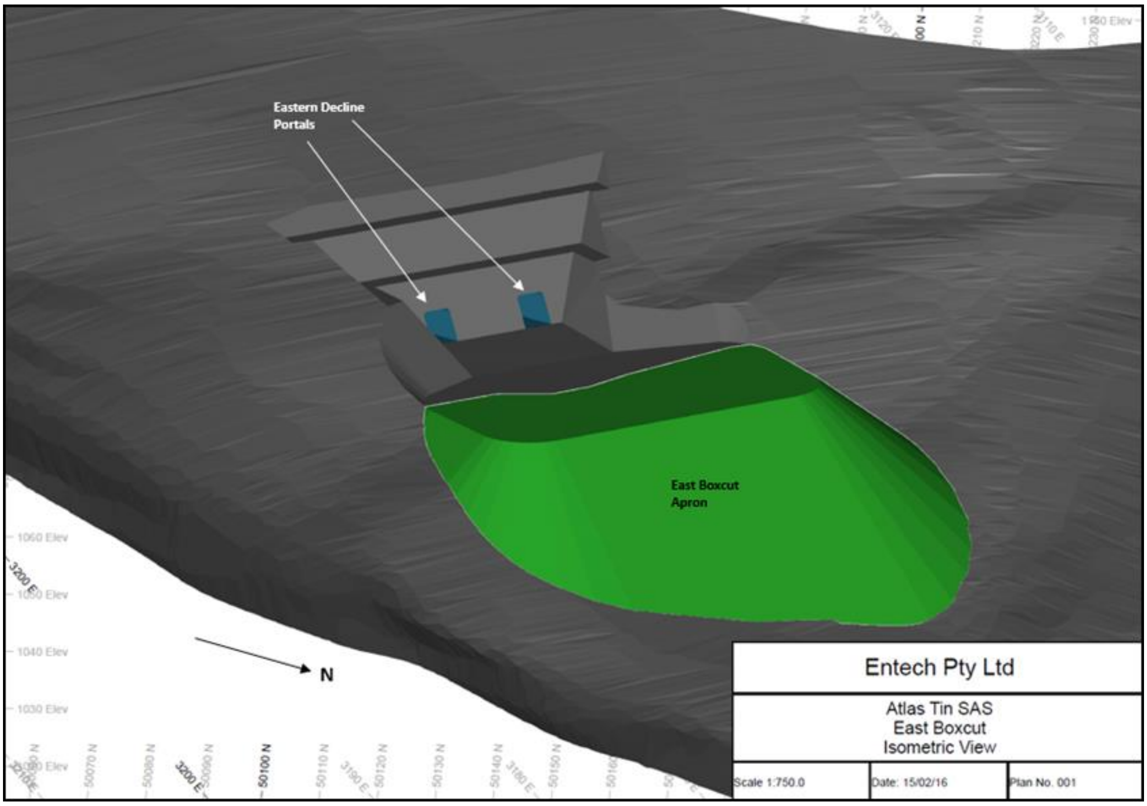


Figure 5-16 Isometric view of eastern boxcut and apron (Looking SW)

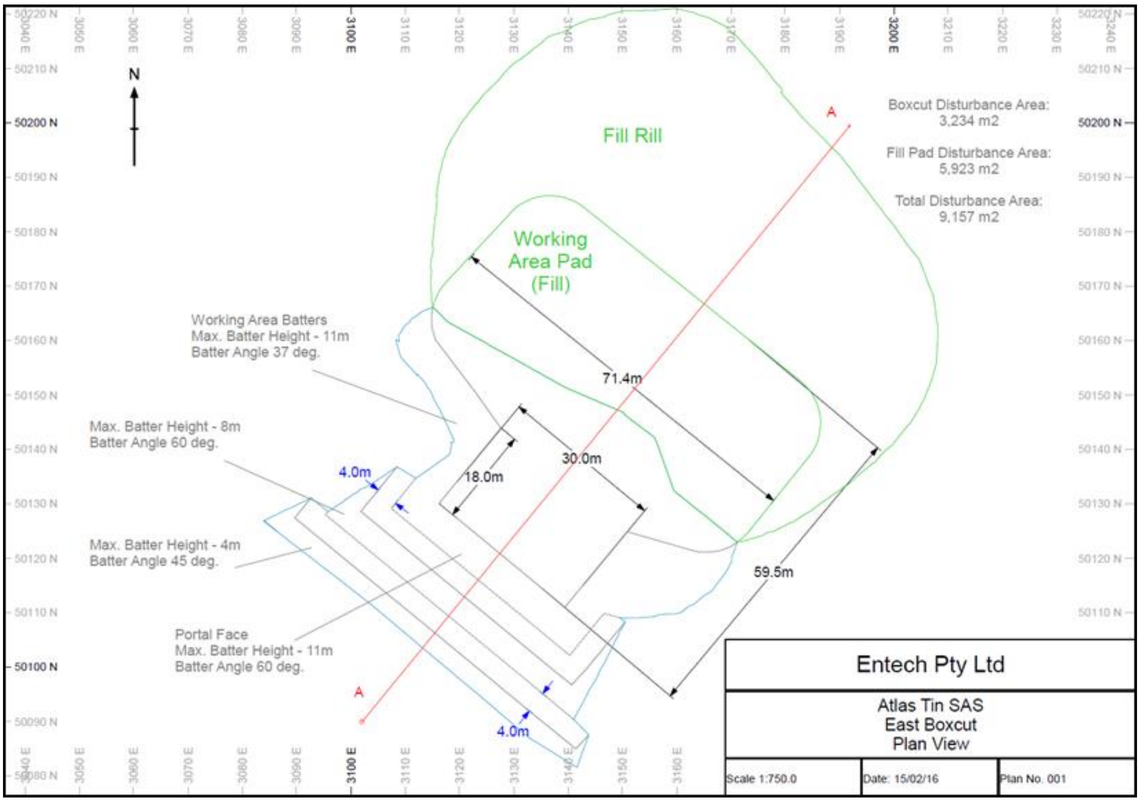


Figure 5-17 Plan view of eastern boxcut and apron

Isometric and plan views of the updated waste dump design are shown in Figure 5-18.

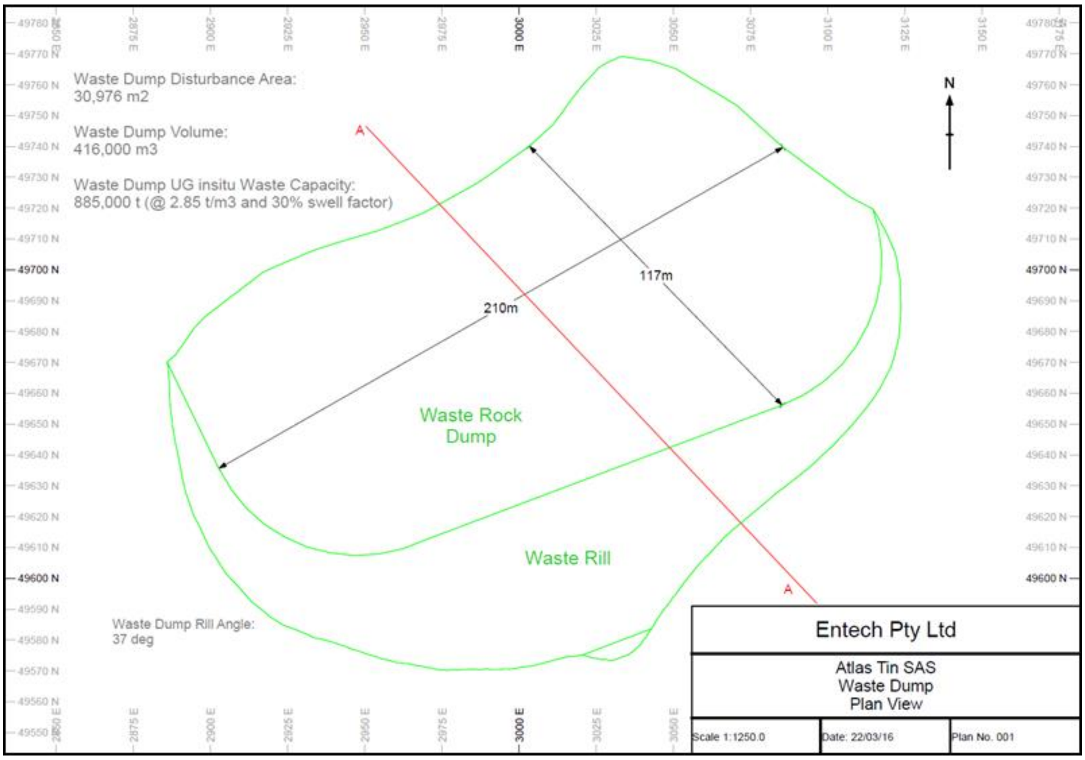


Figure 5-18 Plan view of surface waste dump

The ROM pad design has been revised from previous studies with the footprint maintained to allow suitable storage area for ore classification fingers and working room for trucks and loaders.

The final physicals for construction of the mining surface infrastructure are presented in Table 5-12.

Table 5-12 Mining surface infrastructure construction physicals

Structure	Disturbance Area (m^2)	Cut Volume (m^3)	Fill Volume (m^3)
Central boxcut and apron	26,993	49,165	199,703
Eastern boxcut and apron	9,157	12,139	36,530
Surface waste dump	30,976	-	416,000

5.9 Mining Factors

5.9.1 Mining Dilution

Stope dilution assumptions were determined using hydraulic radius and modified stability number values and cross-checked with mines in similar rock mass conditions. The results provided recommended dilution assumptions of <0.5 m on each footwall (FW) and hangingwall (HW) contact. A dilution thickness of 0.5 m on each contact was used for design purposes. The dilution volumes were estimated by expanding each individual stope section to conform to the dilution recommendations, i.e. 0.5 m of dilution on each HW and FW contact. Dilution grades were then determined by the Resource material contained within the dilution skin.

In the Central Zone areas where stopes are assumed to be blasted against fill and where loaders will be bogging stopes on fill floors, 0.5 m of dilution at zero grade on each fill contact was applied (Figure 5-19 and Figure 5-20). Stopes being mined underneath CRF stopes at the top of panels were assumed to be brought up short so that CRF sills will not be undercut.

Ore development has been assumed to have no dilution.

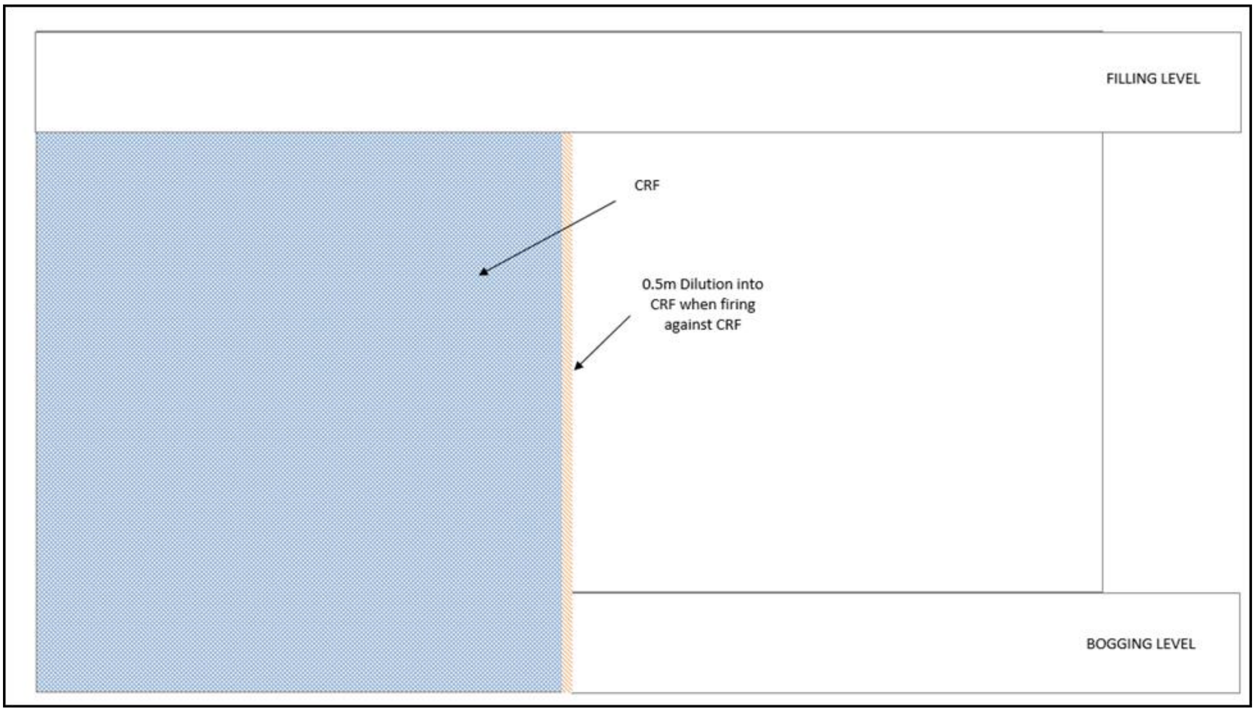


Figure 5-19 Dilution applied due to firing against CRF

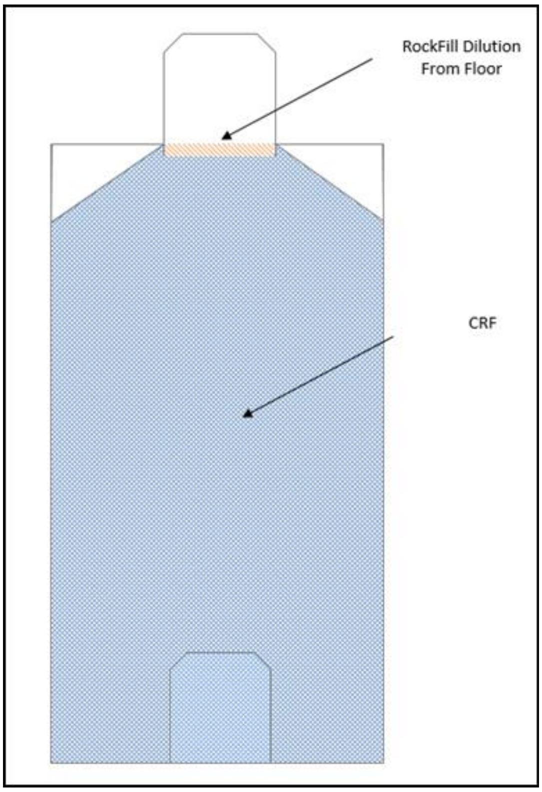


Figure 5-20 Dilution applied due to over-digging of CRF floors

The average stope dilution for the using these parameters is 18%. A histogram of the dilution applied over 5 m stope sections is shown in Figure 5-21.

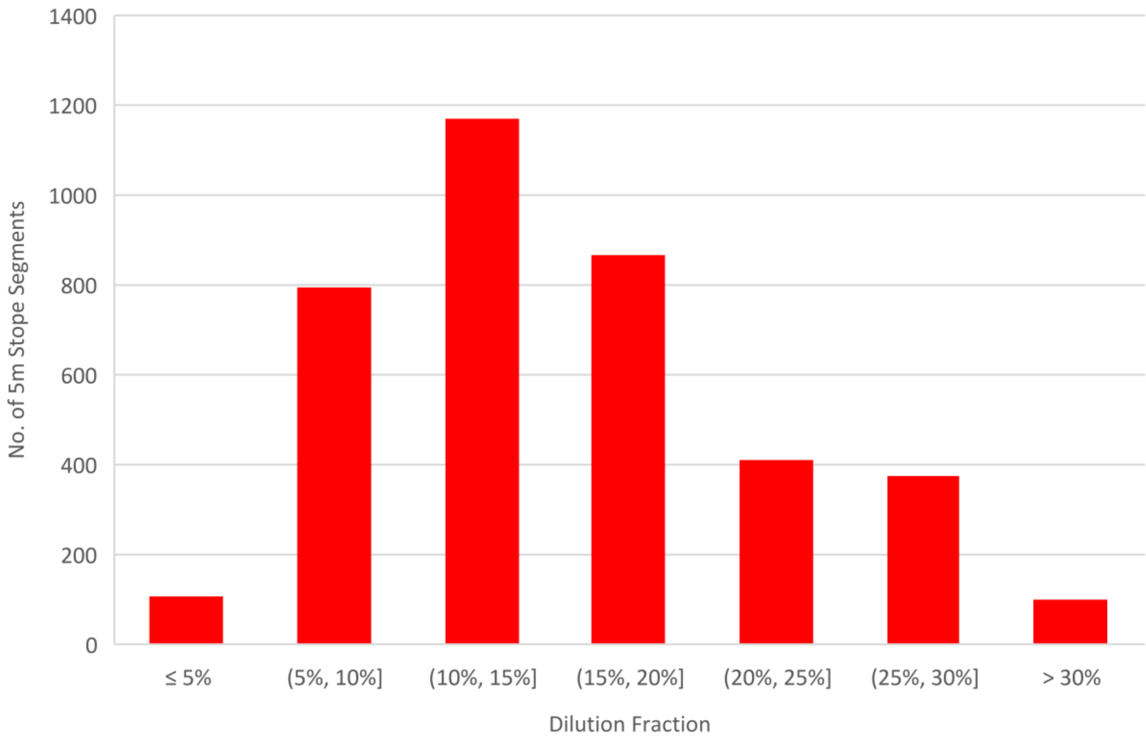


Figure 5-21 Dilution Histogram

5.9.2 Stope Recovery

All stopes have a 95% mining recovery factor applied to account for losses due to drilling and blasting and stope bogging.

Where stopes are wider than 5 m in the bottom-up Central Zone areas, additional ore losses have been applied due to the difficulty of filling the top corners of stopes completely when loading CRF from the top drive together with the geotechnical recommendation to leave pillars to maintain top access walls (Figure 5-22). These recovery losses are based on an assumed rill angle of 38°. The average ore loss in these wide stopes due to leaving the top corners equates to an additional 2.5%.

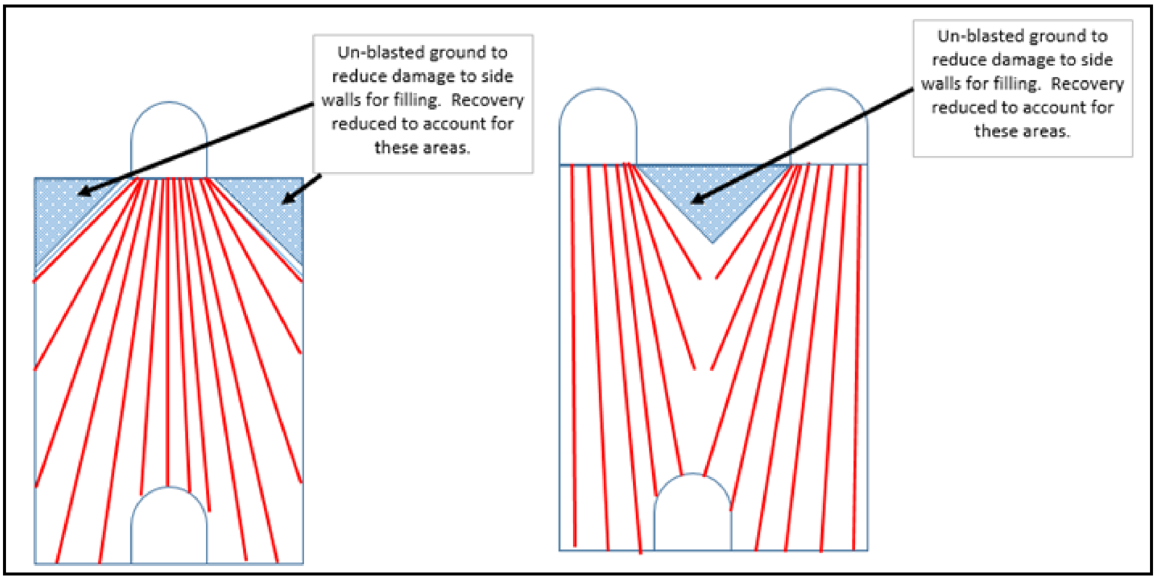


Figure 5-22 Ore recovery loss in wide bottom-up stopes

A histogram depicting stope recovery distributions is presented in Figure 5-21. This shows that for the large majority of stope segments the stope recovery factor is 90% to 95%.

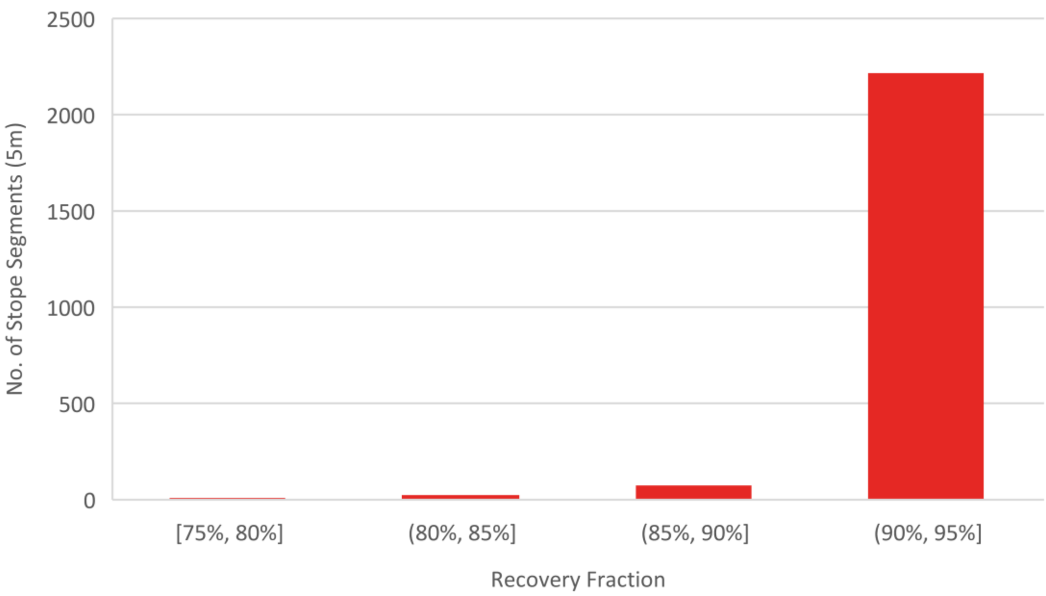


Figure 5-23 Achmmach mine plan stope recovery histogram

Ore recovery for development was assumed to be 100%.

Half-height sill pillars (i.e. pillar ore losses of 50%) were designed for the bottom-up sequence areas where stoping was being undertaken directly underneath filled stopes.

Pillar losses in the top-down open stoping areas are based on the geotechnical recommendations outlined in Section 4.12.2. Pillars are placed in individual stope sections which allowed final tonnage recoveries to be adjusted accordingly. The total Resource left in-situ (i.e. prior to modifying factors being applied) due to pillars is summarized in Table 5-13. This represents a global resource loss to pillars of 10%, which equates to 9% of tin metal on top of the 95% mining recovery.

Table 5-13 Resource loss due to pillars

Pillar	Resource Pillar Ore Tonne Loss (t)	Resource Pillar Loss Grade (% Sn)	Resource Pillar Tin Metal Loss (t Sn)
Rib pillar	353,810	0.79	2,798
Sill pillar	316,077	0.79	2,492
Total pillar losses	669,886	0.79	5,290

5.10 Blasting

Development and stope blasting will be carried out primarily using ANFO with non-electric or signal tube detonators and emulsion primers for development and PETN primers for stoping. Wet holes will be charged with packaged emulsion cartridges.

Development and production charging will be performed either by the integrated tool carrier fitted with a specialised charging basket or by the dedicated Normet charge wagon.

An analysis of the drill and blast design was performed for a range of stope sizes to determine the drill yield. Indicative stope ring designs were generated for stopes from 3 m width to 10 m width, and drill yields applied accordingly, based on each stope section width. The average drill yield determined was 8.3 t per metre of drill hole, assuming 76 mm diameter holes.

5.11 Mining Productivities

The productivities for mine scheduling are based on the performance expected form a competent underground mining contractor. The schedule was generated using the productivity rates detailed in Table 5-14. The rates were estimated by Entech based on experience and first principles calculations.

Table 5-14 Key productivity rates and schedule delays

Activity	Unit	Productivity
Lateral jumbo development	m adv/jumbo/month	130 increasing to 250 by mth 3
Max advance in a heading	m adv/heading/week	30
Production drilling	drm/drill/day	220
Stoping (7m^3 loader)	t/loader/day	1,000
Stoping (stope)	t/stope/day	500
Backfill loading (7m^3 loader)	t/loader/day	800
Backfill loading (stope)	t/stope/day	800
Truck haulage	tkm/truck/month	80,000-90,000
Capital vertical development	m adv/heading/d	2
Diamond drilling	drm/rig/day	50
Diamond drilling to development delay	days	18
Development to stoping delay	days	14
Slot rise delays	days	7
CRF cure delay (to adjacent stope)	days	14

Interstitial delays were applied as follows:

- From completion of diamond drilling to commencement of constrained development – 18 days, allowing for core preparation, transport, assaying, geological modelling and mine design;
- From completion of ore drives to commencement of stope drilling in that drive – 14 days, allowing for face sample assaying, geological modelling and production drilling design;
- Stope slot raise delays – 7 days; and
- CRF curing delays – 14 days per CRF filled stope. Entech considers this to be a reasonable curing time based on 5% cement content and the requirement to mine adjacent stopes only. No CRF stopes are undercut in the mine plan.

Trucking productivities used in the mine schedule are presented in Table 5-15. This is based on the actual haul for 50 t articulated underground trucks and requires approximately 450 operating hours per month and a productivity rate of 190-200 tkm/hr.

Table 5-15 Achmmach Trucking Productivity Assumptions

Material	tkm/month/truck
Central portal ore	90,640
Central portal waste	84,777
East portal ore	88,297
East portal waste	88,080

5.12 Cemented Rock Fill

Cemented rock fill (CRF) is a simple stabilised backfill which involves placement of waste rock mixed with cement slurry into the stope void by a loader from a drive at the top of the stope. CRF consists of blasted development waste rock mixed with cement slurry in mixing cuddies underground. The mixing cuddy is located on the level above the stope and can be the level stockpile or an inactive drive. Floors need to be blasted out to create a shallow sump to allow the loader to mix the waste rock and cement slurry adequately.

The proposed CRF method is as follows;

- Trucks will dump waste rock in the mixing cuddy, either directly from a blasted face or back hauled from surface.
- Cement slurry will be batched in a plant on the surface. The mix recipe will depend on the final site assessment, however preliminary a mix design for planning and costing purposes is detailed in Table 5-16. This is based on current practice in West Australian mines.
- Cement slurry will be transported to the mixing level by agitator truck. An 8 m^3 capacity truck is typical and has been assumed for the purposes of productivity rate determination.
- The cement slurry is poured into the mixing cuddy and the loader mixes the rock and slurry.
- The resulting CRF mix is loaded into the stope void by the loader.

Table 5-16 Proposed CRF mix

Item	Unit	Value
CRF cement strength	%	3-5
Water: cement ratio		0.8

Cement density	t/m^3	1.5
Water density	t/m^3	1.0
Rockfill density (30% swell)	t/m^3	2.2
Cement slurry batch volume	m^3	8.0
Cement tonnes in batch	t	5.5
Water tonnes in batch	t	4.4
Rock tonnes in batch	t	100
CRF tonnes/batch	t	110

Schematics of an indicative mixing cuddy assuming use of the level stockpile or an inactive drive are presented in Figure 5-24 and Figure 5-25.

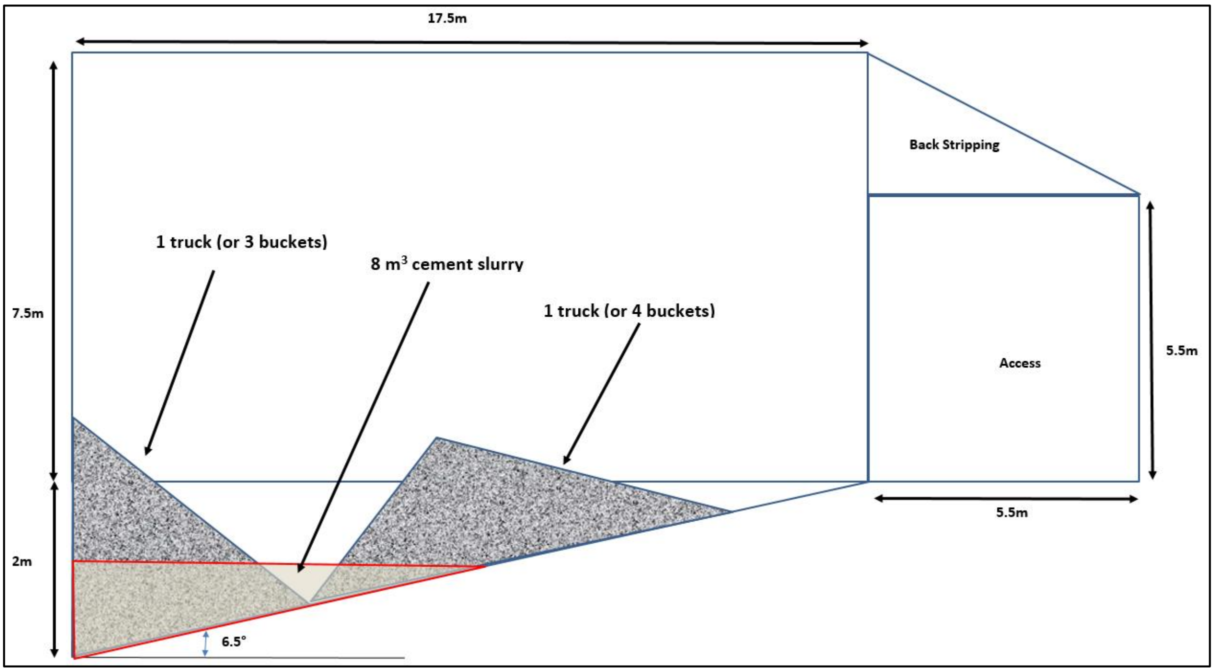


Figure 5-24 Long-section schematic of stockpile mixing cuddy (not to scale)

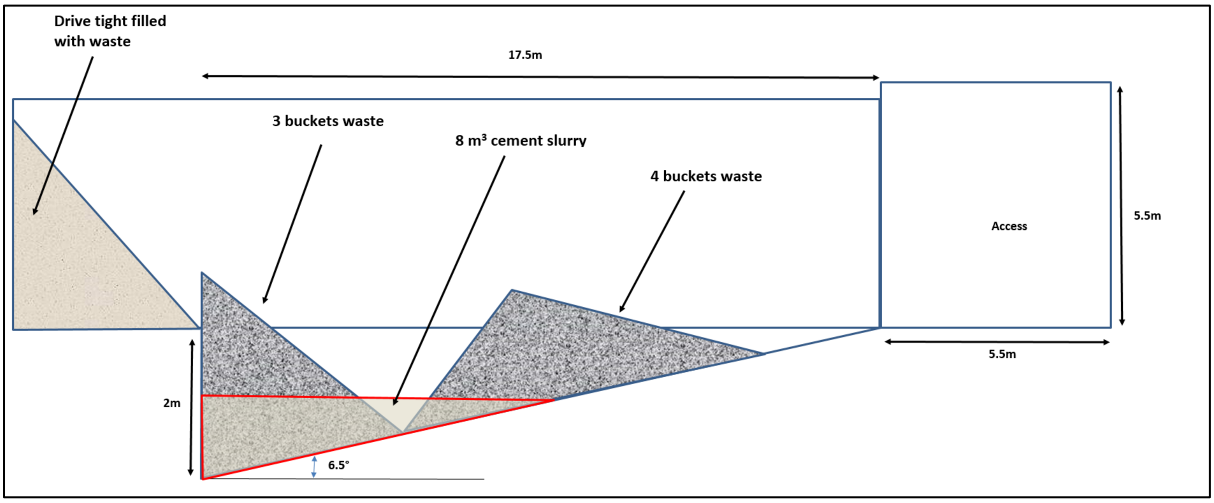


Figure 5-25 Long-section schematic of inactive drive mixing cuddy (not to scale)

Filling productivity has been determined based on the use of 7m^3 sized loaders. Table 5-17 and Table 5-18 outline the calculated daily productivity for loading of cemented rockfill and uncemented rockfill material into stope voids.

Table 5-17 CRF Placement Productivity

CRF Loader Productivity	Value	Comment
Ave. Drive length (m)	150	From EPS
CRF bogger productivity (R2900) t/hr	85	Instantaneous rate from Entech database
Availability	0.9	Entech database
Utilised hours/day	18	Entech database
Potential CRF bogger productivity (R2900) t/d	1,377	

Table 5-18 Uncemented Rockfill Placement Productivity

Rockfill Loader Productivity	Value	Comment
Ave. drive length (m)	150	From EPS
Rockfill bogger productivity (R2900) t/hr	132	Instantaneous rate from Entech database
Availability	0.9	Entech database
Utilised hours/day	18	Entech database
Total rockfill bogger productivity (R2900) t/d	2,138	

The bottleneck activity for CRF productivity is generally the delivery of cement slurry to the mixing level. The productivity estimate for this activity has been determined assuming the use of 8 m^3 capacity underground agitator trucks. The basis for the productivity estimate is presented in Table 5-19.

Table 5-19 Cement Slurry Placement Productivity

Cement Slurry Productivity	Value	Comment
Ave. Haul distance from surface (km)	1.5	From EPS lookup tables
Surface haul distance (km)	0.2	
Speed surface (km/h)	10	OEM Specs for WR820
Speed loaded down ramp (km/h)	8	OEM Specs for WR820
Speed up ramp (km/h)	8	OEM Specs for WR820
Surface load time (min)	10	
Ug dump time (min)	46	OEM Specs for WR820 pump, 147 L/min
Cycle time (min)	82	
Availability	85%	
Utilised hours/day	18	
Cement agitator loads/day	11	
Cement agitator capacity (m^3)	8	
Cement agitator productivity (m^3/d)	88	
Cement slurry productivity (CRF t/d)	1,202	

The productivity for all backfilling was set to maximum 800 t/d for the mining schedule. This rate is below theoretical target levels and allows greater flexibility in the overall production sequence and for scheduling conflict between truck haulage for ore and waste.

CRF was selected as the preferred fill method rather than paste backfill for the following reasons:

- The lateral extents of the Achmmach deposit would require considerable pumping infrastructure to deliver paste to all stoping areas;
- CRF is more flexible as filling can occur concurrently in separate mining areas (paste filling typically only occurs in a single location at a time);
- CRF requires minimal capital infrastructure as opposed to pastefill which requires a product mixing plant coupled to the ore processing plant and extensive reticulation systems;
- The risk of liquefaction is removed and other pastefill associated technical risks (e.g. fill characteristics, fill scheduling) are minimised or eliminated from the mine plan;
- Technical complexity and project risk is reduced as CRF is a simpler filling method;
- Curing and re-entry periods for CRF are typically shorter; and
- The method stores most waste generated by the mine underground, reducing required footprints for surface waste dumps.

The disadvantages of CRF as opposed to pastefill are as follows;

- CRF quality control is more difficult than for pastefill and so CRF can be a less consistent product;
- CRF requires increased diesel equipment utilization, including loading and cement transportation vehicles, with associated traffic and ventilation effects.

The quality disadvantage of CRF has been partially overcome by eliminating the requirement to work directly beneath fill (in-situ sill pillars have been assumed to be left when stoping is being undertaken directly underneath CRF filled stopes).

The use of pastefill remains an option for the project should the tin price significantly increase. The benefit would be the higher rates of ore extraction available with use of paste which under a high commodity price environment would outweigh the capital and additional operating costs. Figure 5-26 illustrates the proposed mining method (note that the central access shown in the top level is omitted from the lower levels in the diagram for clarity).

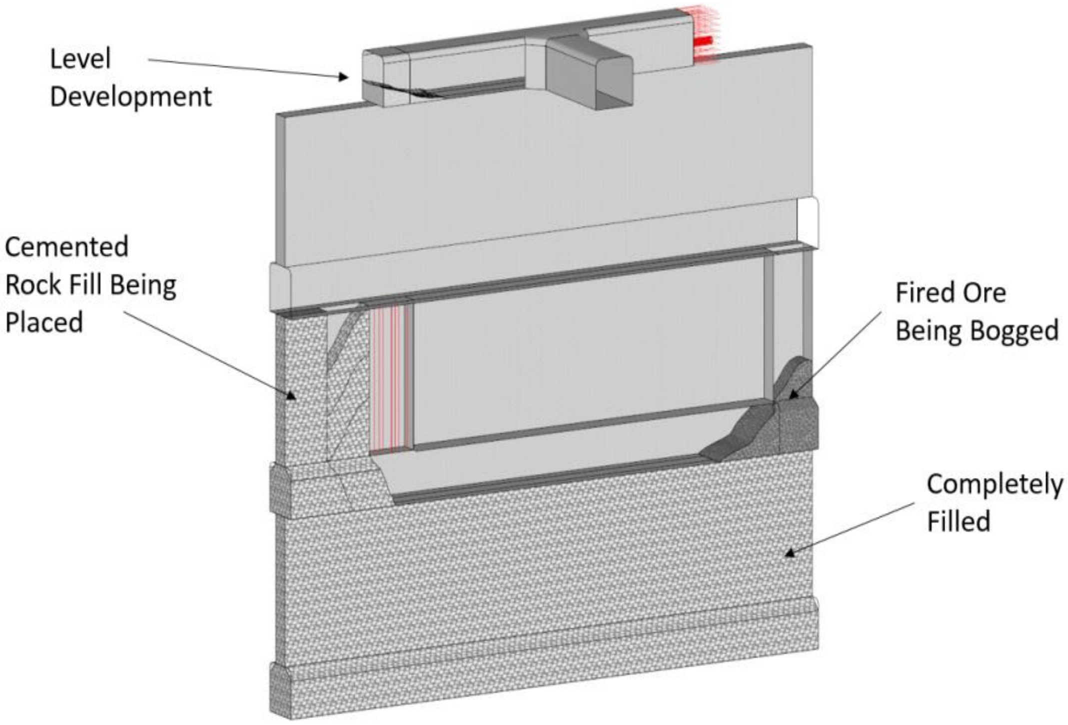


Figure 5-26 Bottom-up CRF mining method

The backfill plan was reviewed and adjusted such that stopes that were considered not practical to backfill (e.g. at the top of a stoping block and without top access drives) were left unfilled thus reducing mining costs.

The selected mining method allows for a proportion of the stope to be filled using uncemented rock, if a suitable cemented portion is retained on the boundaries that will be required to stand up against proximal blasted voids. The proportion of fill required to be cemented was estimated for the CRF stopes based on a 45° rill angle and a minimum cemented pillar thickness of 5 m against void boundaries, Figure 5-27. Stopes that will not have a void boundary are assumed to be filled entirely with uncemented waste.

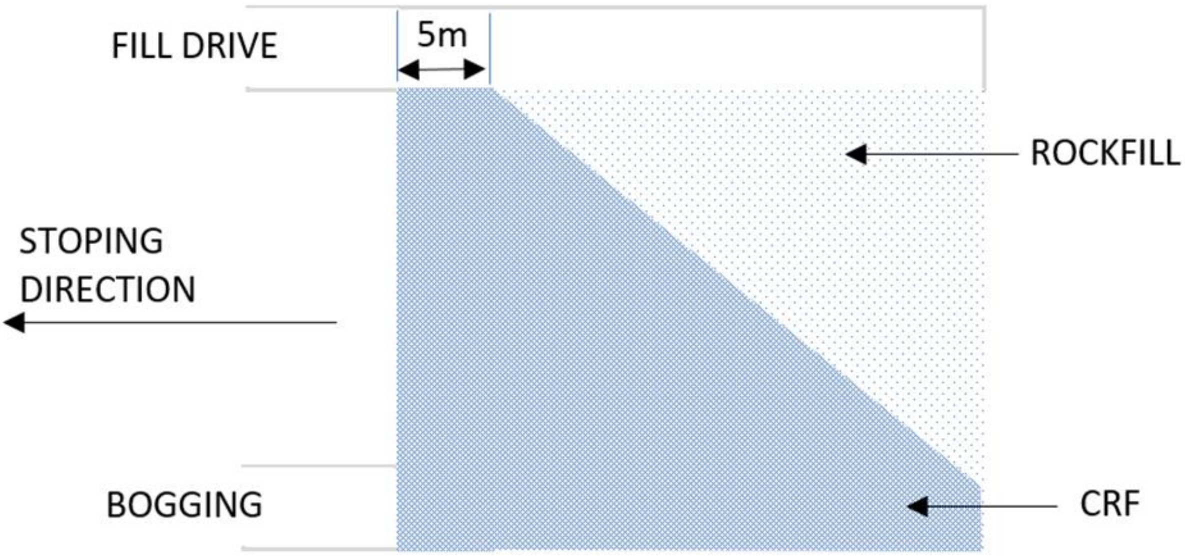


Figure 5-27 Section of indicative stope fill geometric proportions

The proportion of uncemented fill in the mine plan is 56% of total fill, with the remaining 44% being CRF. Total percentage of filled stopes within the stoping envelope (by mass) is 37%.

Cement usage for cemented fill has been assumed to be 5% by weight as recommended in the geotechnical section, i.e. 1 t of cemented backfill will contain 50 kg of cement.

5.13 Stope Economics

Following completion of the stope design, development design, drill and blast review, mining factor assumptions and geotechnical review, the economics of the final stoping blocks were reviewed using costs from the detailed cost model and the results of a request for quotation (RFQ) process, together with revenue factors provided as shown in Table 5-20.

Table 5-20 Assumptions Used for Economic Analysis of Stoping Areas

Item	Unit	Value
Revenue Factors		
Royalty	%	3%
Realized Tin Price	US$	$19,400
Tin Metallurgical Recovery	%	70%
Cost Factors		
Decline Development	$/m	$2,136
Capital Lateral Development	$/m	$2,136
Operating Waste Development	$/m	$1,828
Operating Ore Development	$/m	$1,828
Vent Rise-Raisebore	$/m	$5,454
Vent Rise-VCR	$/m	$1,659
Escapeway	$/m	$8,968
Waste Haulage	$/tkm	$1.52
Ore Haulage	$/tkm	$1.84
Stoping	$/t	$17.58
Backfill	$/t	$1.89
Mine Overheads	$/t	$6.23
Processing	$/t	$18.66
Administration	$/t	$5.08
Con Transport	$/t	$6.66

Those stoping blocks or sections which were sub-economic were removed from the mine plan as were stopes that required significant development to access small tonnages.

5.14 Ore Reserve & Mining Schedules

5.14.1 Introduction

The mine plan schedule was developed using EPS® software. The mine plan for a production ramp-up from 500 ktpa in the first year to 750 ktpa of ore delivery to the processing plant as quickly as possible. This required rapid establishment of concurrent mining areas to provide the feed.

5.14.2 Ore Reserve

The ore delivery is presented in Figure 5-28.

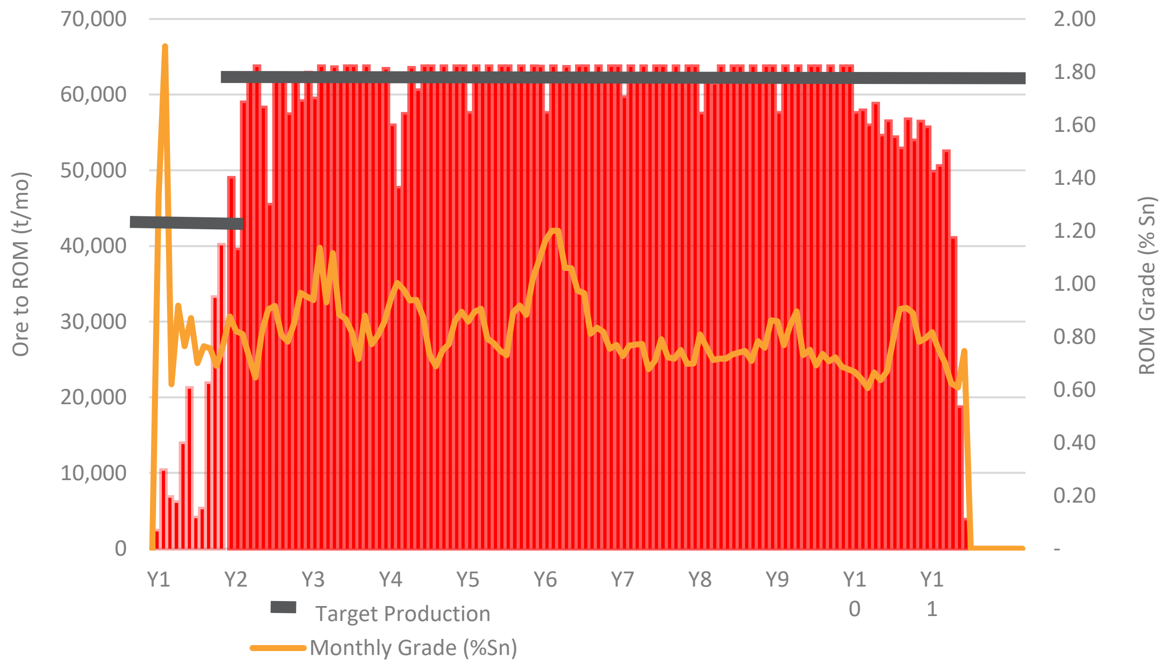


Figure 5-28 Mine plan ore delivery schedule (showing average monthly ROM target)

5.14.3 Mining Sequence

The mining sequence by zone within the overall mine model is illustrated in Figure 5-29. The Central Upper (also referred to as the Fez Zone) represents an opportunity to extract ore at the end of the presently designed mine life and was not scheduled LOM plan.

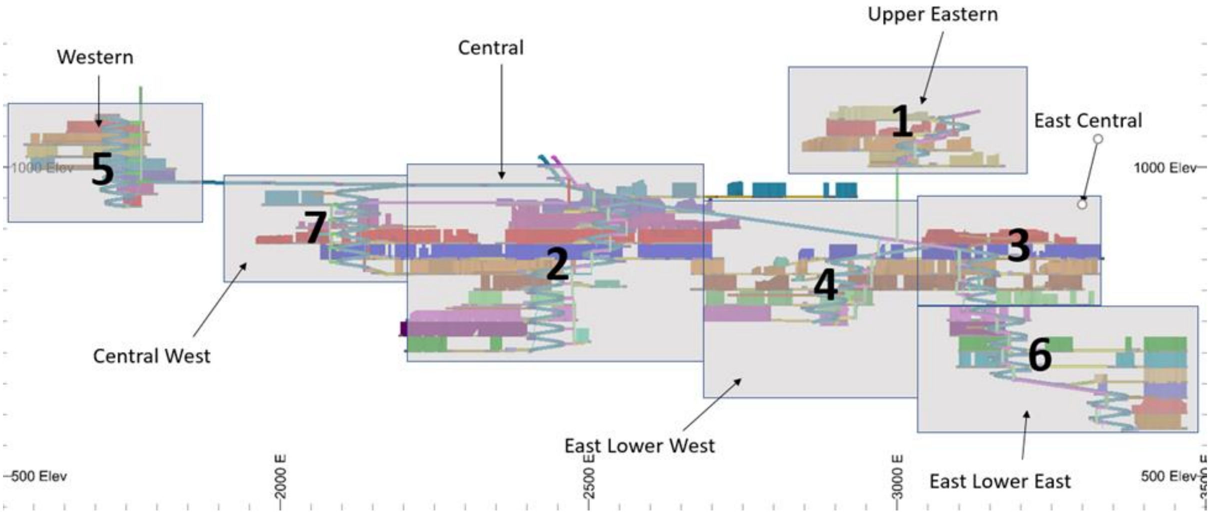


Figure 5-29 Achmmach mine plan mining sequence (long section looking North)

Development in both the Central and Eastern Declines commences simultaneously from Month 1. The short decline development into the Eastern Zone will provide early delivery of ore to the ROM pad, allowing mill commissioning to take place while the Central Zone is still under development. The Eastern Zone will provide about 40,000 t of ROM ore for 4-6 months prior to the delivery of first ore from the larger, more productive Central Zone.

The following extracts from mine sequence animations shown below illustrate the evolution of the mine.

Development in both the Central and Eastern Declines commences from Month 3 as shown in Figure 5-30.

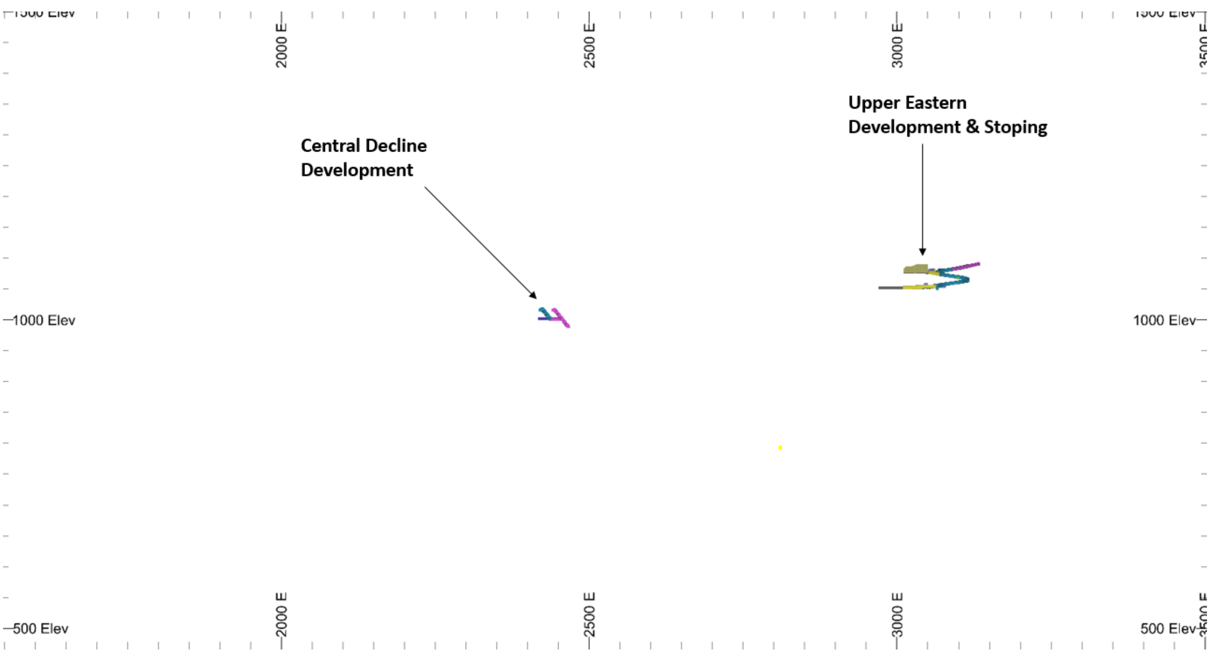


Figure 5-30 Mine schedule month 3 (long-section looking north)

The first stope is mined in the Central Zone in Month 17. The Upper Eastern zone is almost completed, and the Western and Eastern Zone accesses have been commenced at this point as shown in Figure 5-31.

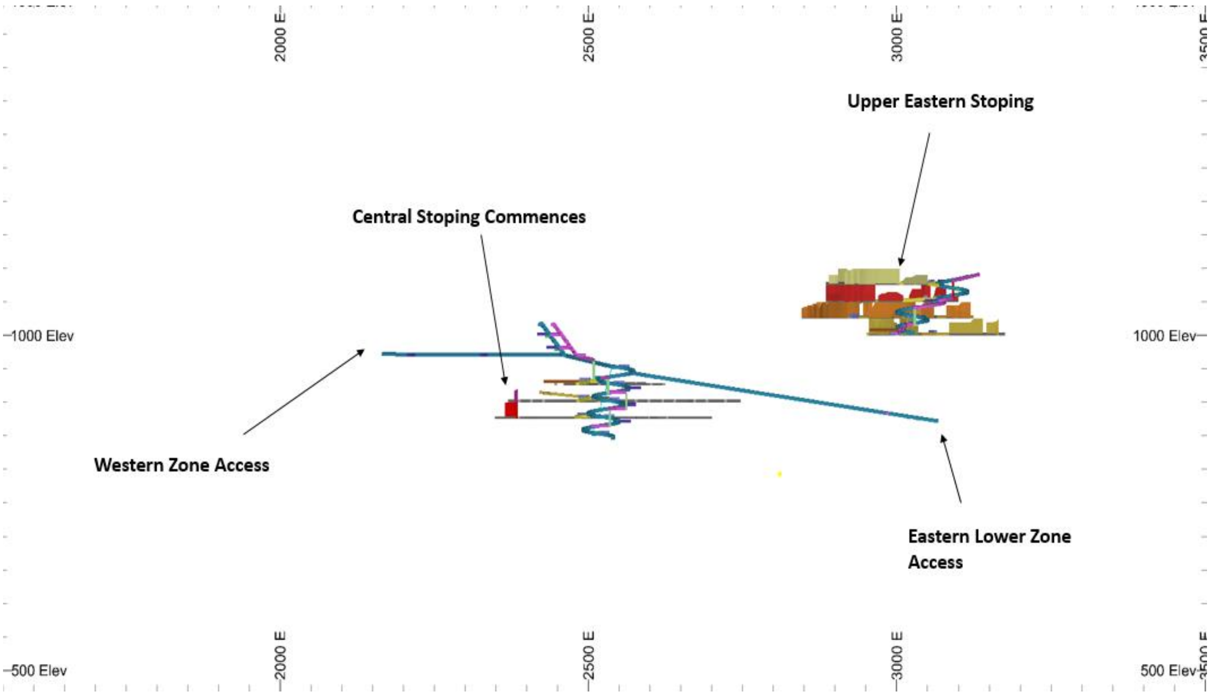


Figure 5-31 Mine schedule month 17 (long-section looking north)

The ventilation and escapeway raise is developed between the Upper and Lower Eastern Zones in Month 36. Development commences in the Lower Eastern Zone at that point as shown in Figure 5-32.

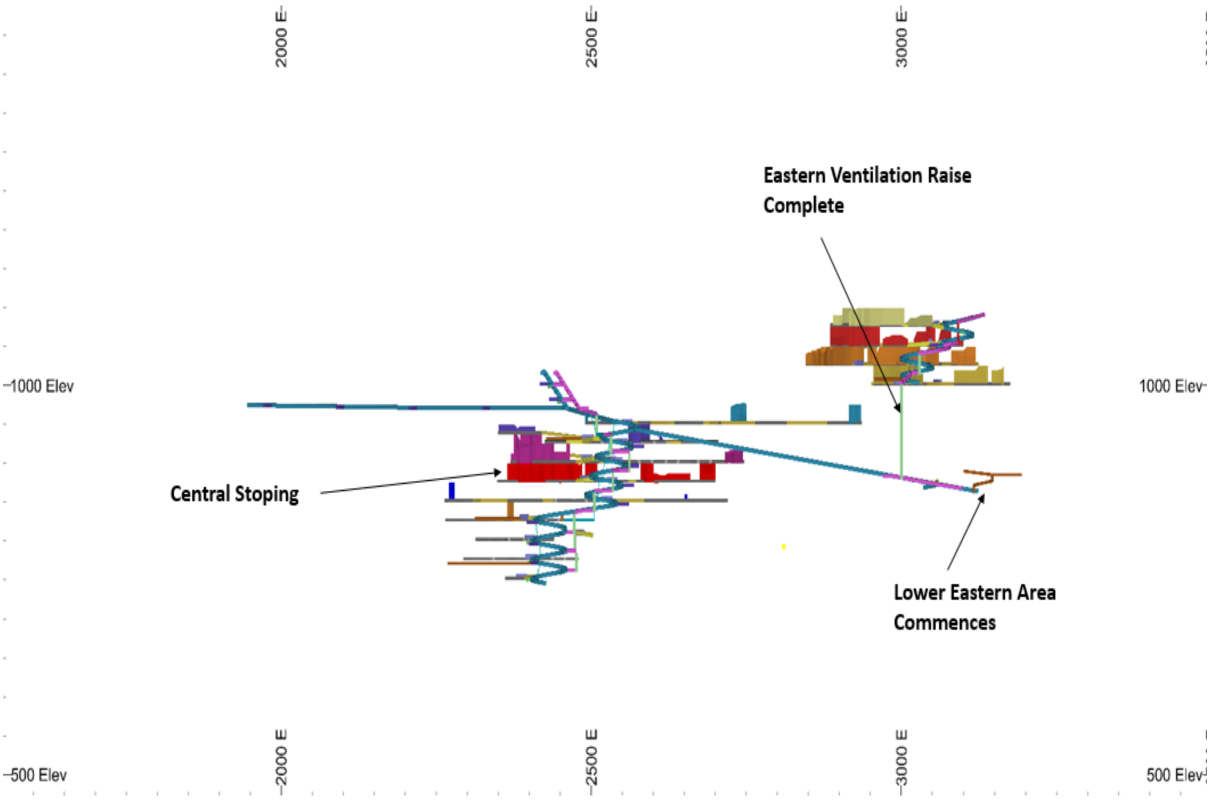


Figure 5-32 Mine schedule month 36 (long section looking north)

Stoping in the Western Zone commences in Month 44. Development in the Central Zone is completed at this point, with jumbo resources concentrated in the Lower Eastern Zone and Western Zone. This is illustrated in Figure 5-33.

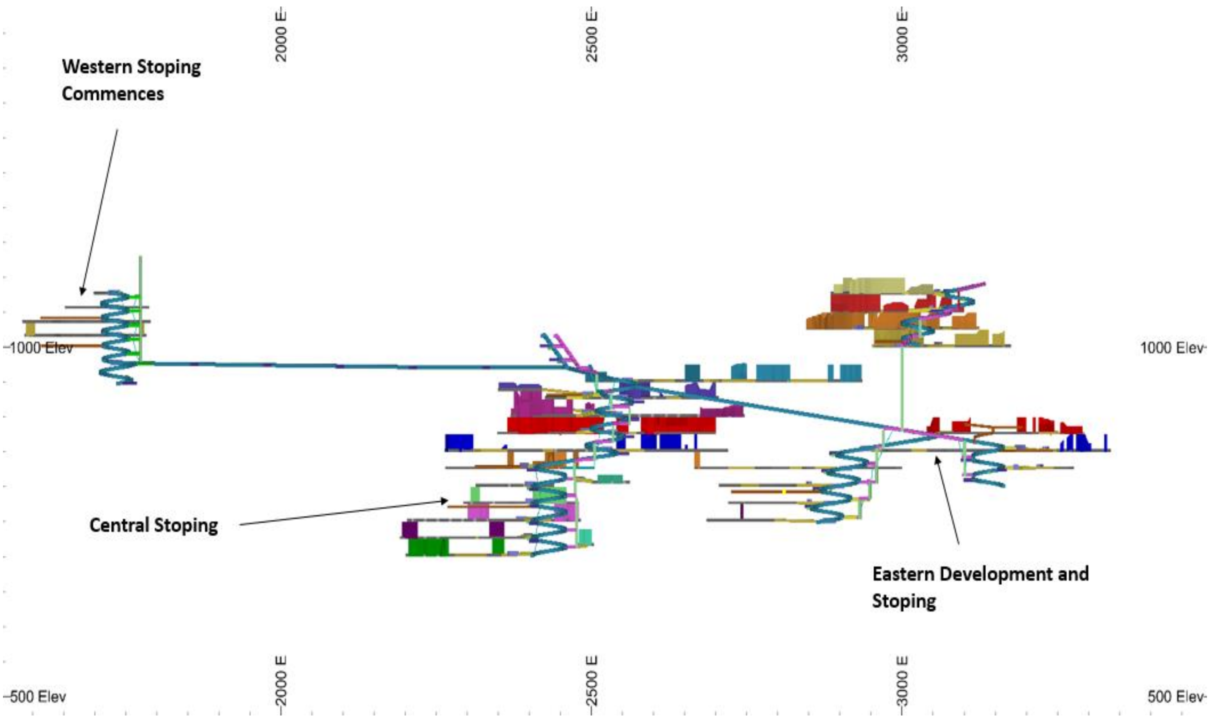


Figure 5-33 Mine schedule month 44 (long section looking north)

At Month 77, stoping commences in the lower Eastern area as shown Figure 5-34.

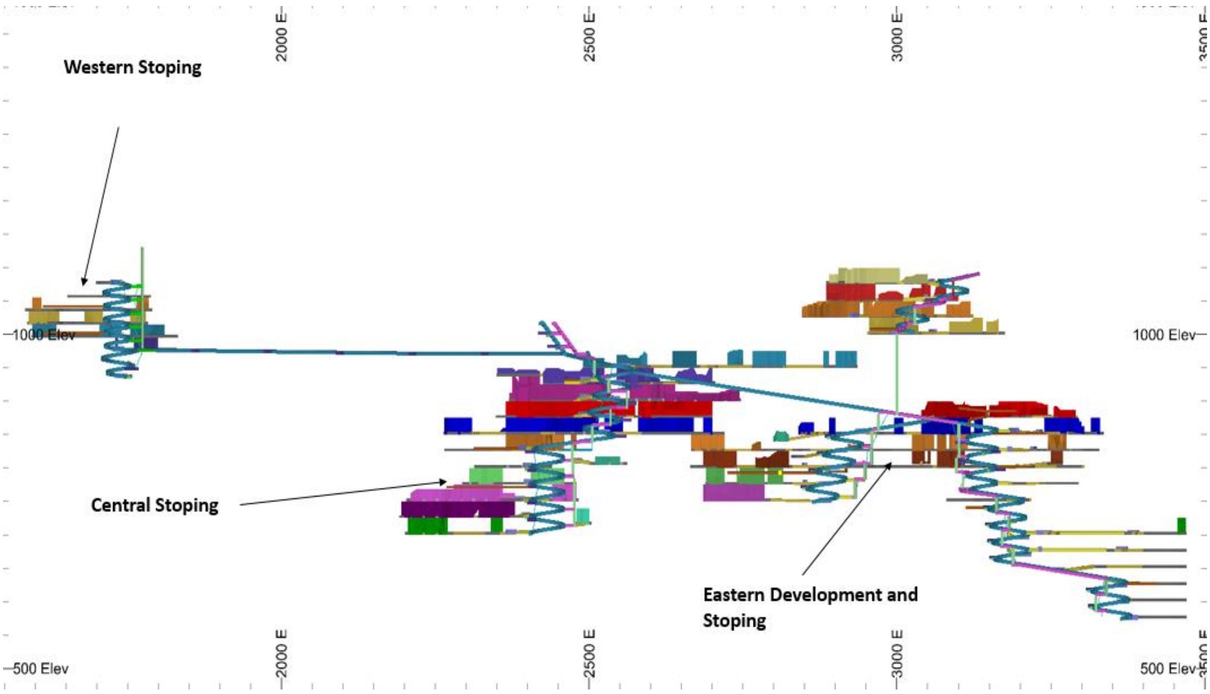


Figure 5-34 Mine schedule month 77 (long section looking north)

Stoping commences in the Central West area in Month 94 as shown in Figure 5-35.

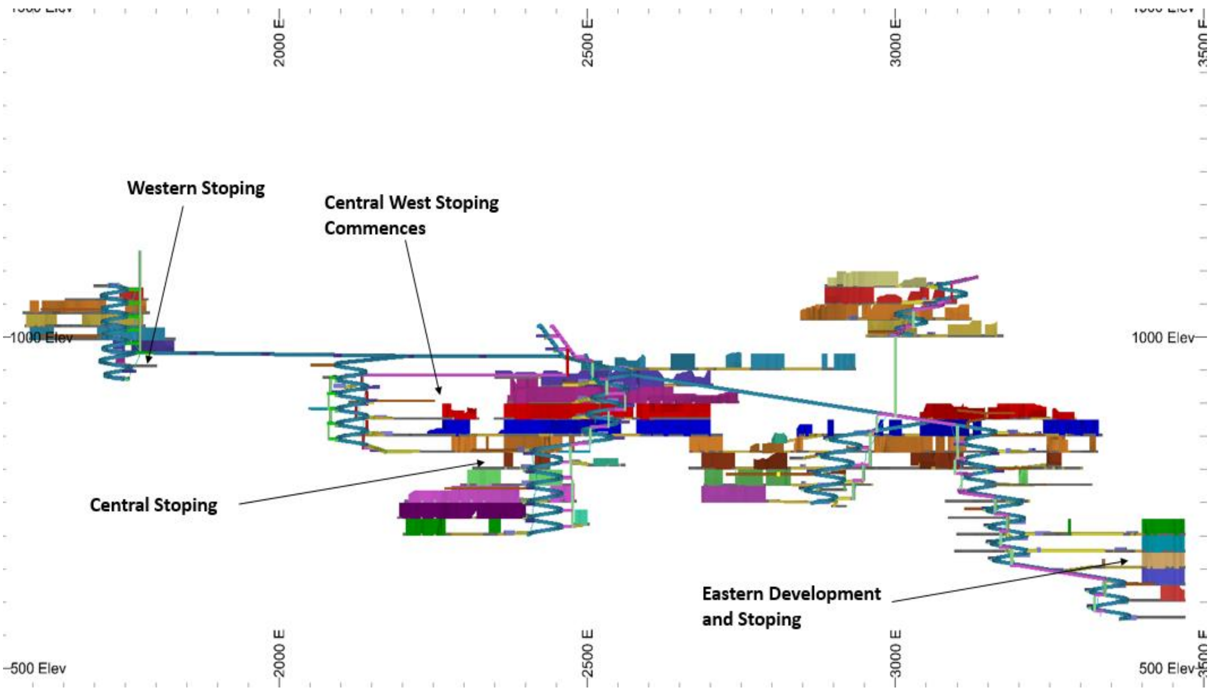


Figure 5-35 Mine schedule month 94 (long-section looking north)

The final years of the schedule involve completion of the bottom-up Central Zone and top-down Eastern, Central Western and Western Zones. The final mine plan at the end of the mine life is presented in Figure 5-36.

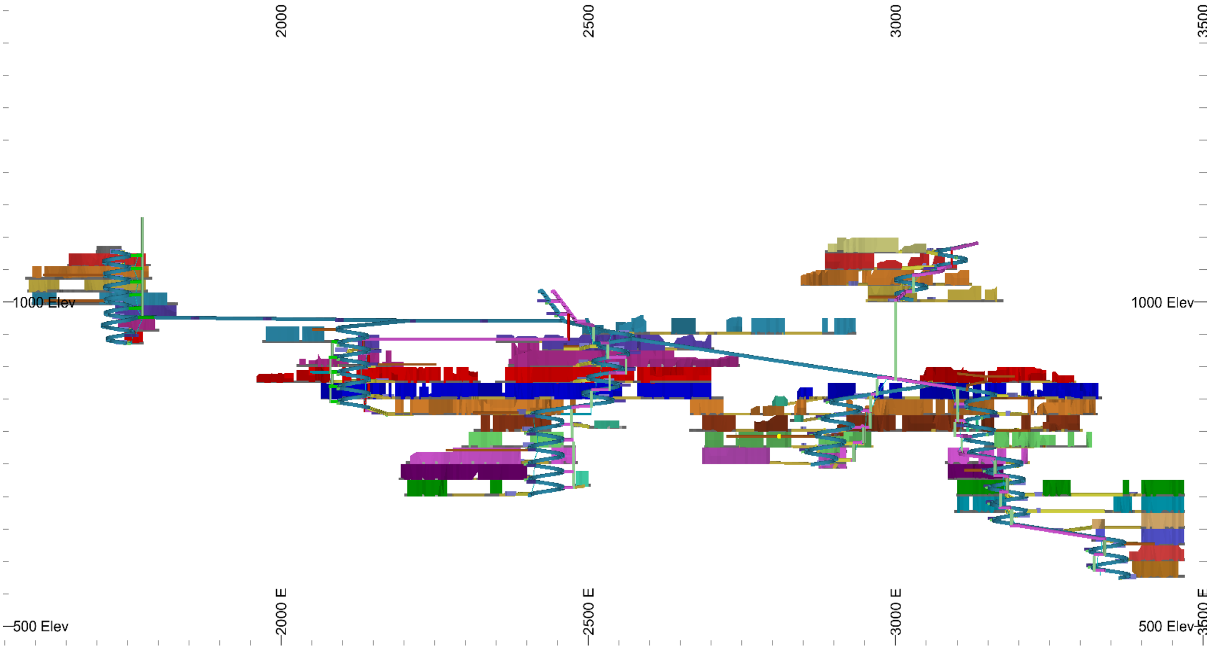


Figure 5-36 Mine schedule month 127, end of mine life (long section looking north)

Table 5-21 Mine schedule physicals

Achmmach Mine Plan: Physicals	FY 2021	FY 2022	FY 2023	FY 2024	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	Total
Operating development (m)	1,329	5,887	3,890	3,377	1,640	1,596	1,686	1,755	1,389	234	-	22,783
Capital development (m)	3,837	3,288	4,077	3,221	1,531	2,247	1,956	1,312	61	-	-	21,529
Total lateral development (m)	5,165	9,175	7,966	6,598	3,171	3,843	3,642	3,067	1,450	234	-	44,312
Production drilling (m)	13,292	49,276	69,594	72,273	81,769	79,042	80,663	73,483	77,546	69,150	27,326	693,414
Waste tonnes (t)	335,906	319,583	381,933	343,617	141,836	205,276	175,742	124,491	17,945	2,264	-	2,048,591
Waste hauled (tkm)	209,547	362,748	625,551	618,434	298,301	573,677	237,257	202,418	16,683	298	-	3,144,914
Ore hauled (tkm)	79,885	658,294	1,017,289	1,097,202	1,103,499	1,172,852	1,247,751	1,371,752	1,298,423	1,206,816	433,331	10,687,094
Cemented backfill (t)	-	62,142	140,231	171,274	171,666	133,022	109,872	146,413	84,078	-	-	1,018,698
Unconsolidated backfill (t)	-	53,500	130,646	100,851	112,125	147,115	147,193	57,456	106,915	20,090	-	875,890
Development ore tonnes (t)	59,814	296,260	193,614	125,730	75,007	71,954	82,237	86,836	71,910	13,182	-	1,076,543
Development Sn grade (%)	0.69%	0.80%	0.78%	0.82%	0.58%	0.64%	0.57%	0.57%	0.67%	0.66%	0.00%	0.72%
Sn metal tonnes (t)	415	2,358	1,512	1,028	432	461	466	496	480	87	-	7,736
Stope ore tonnes (t)	104,701	381,286	555,312	600,100	674,408	677,346	669,231	660,390	677,505	665,088	271,433	5,936,800
Stope Sn grade (%)	0.92%	0.85%	0.94%	0.86%	0.89%	1.03%	0.76%	0.76%	0.78%	0.74%	0.73%	0.84%
Stope Sn metal (t sn)	966	3,222	5,243	5,188	6,004	6,978	5,090	5,035	5,296	4,895	1,993	49,910
Total ore tonnes (t)	164,515	677,546	748,926	725,830	749,415	749,300	751,468	747,226	749,415	678,270	271,433	7,013,344
Sn grade (%)	0.84%	0.82%	0.90%	0.86%	0.86%	0.99%	0.74%	0.74%	0.77%	0.73%	0.73%	0.82%
Sn metal tonnes (t)	1,381	5,580	6,755	6,215	6,436	7,439	5,556	5,531	5,777	4,982	1,993	57,645

The lateral development schedule is presented in Figure 5-37.

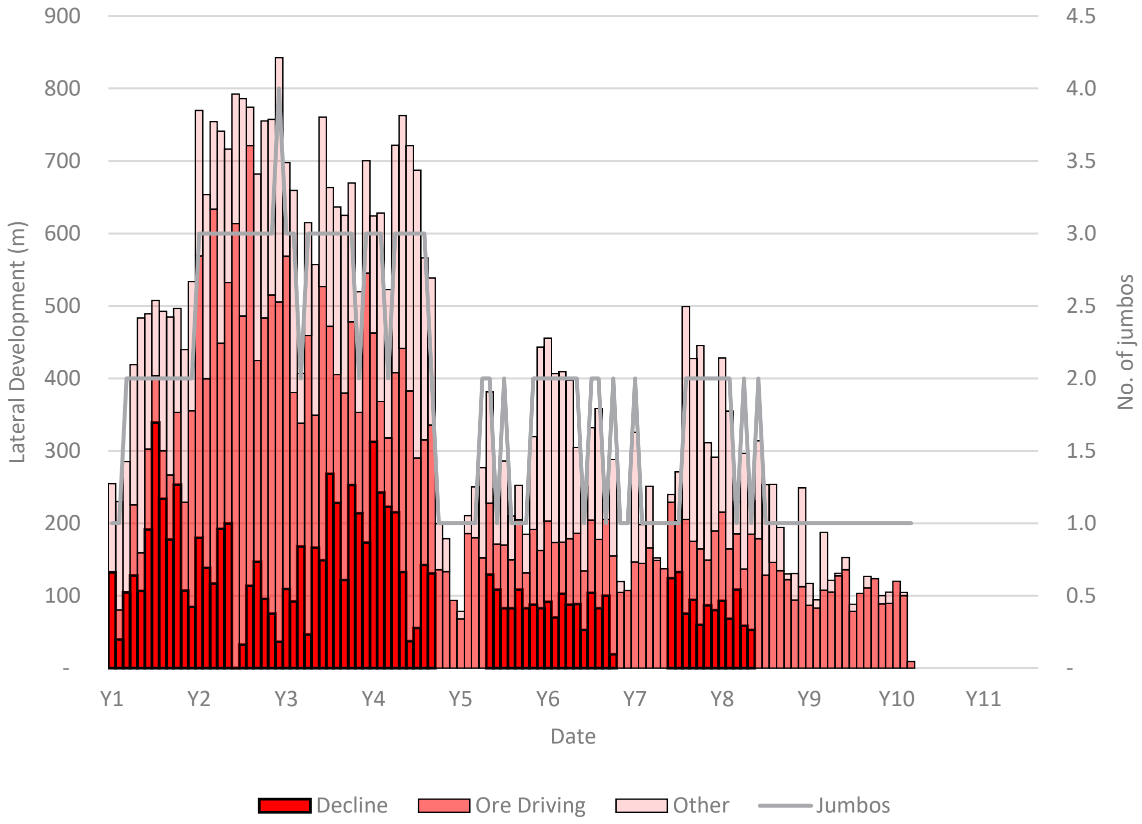


Figure 5-37 Mine plan lateral development schedule

The ore delivery profile is presented in Figure 5-38.

Figure 5-38 Development/stope ore split

The longhole drilling requirements of the schedule are illustrated graphically in Figure 5-39.

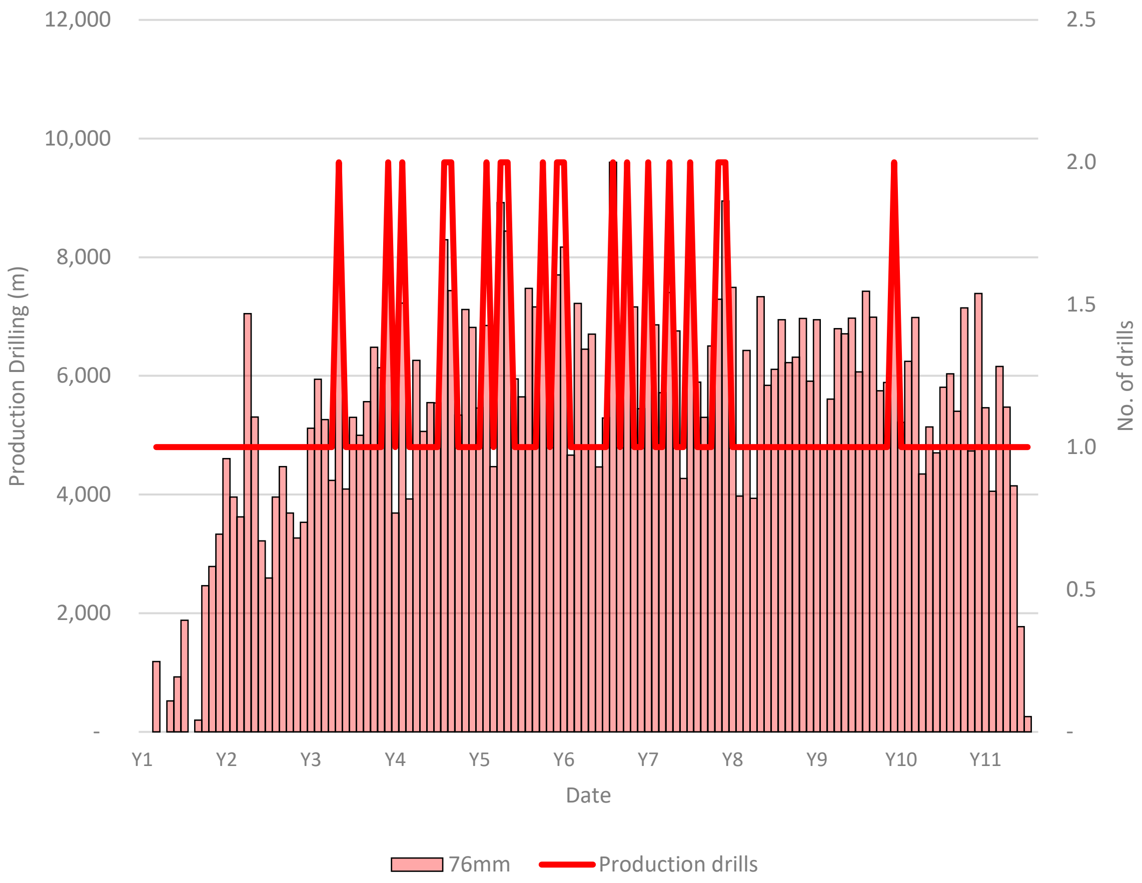


Figure 5-39 Longhole drilling schedule

The haulage requirements of the schedule are illustrated in Figure 5-40.

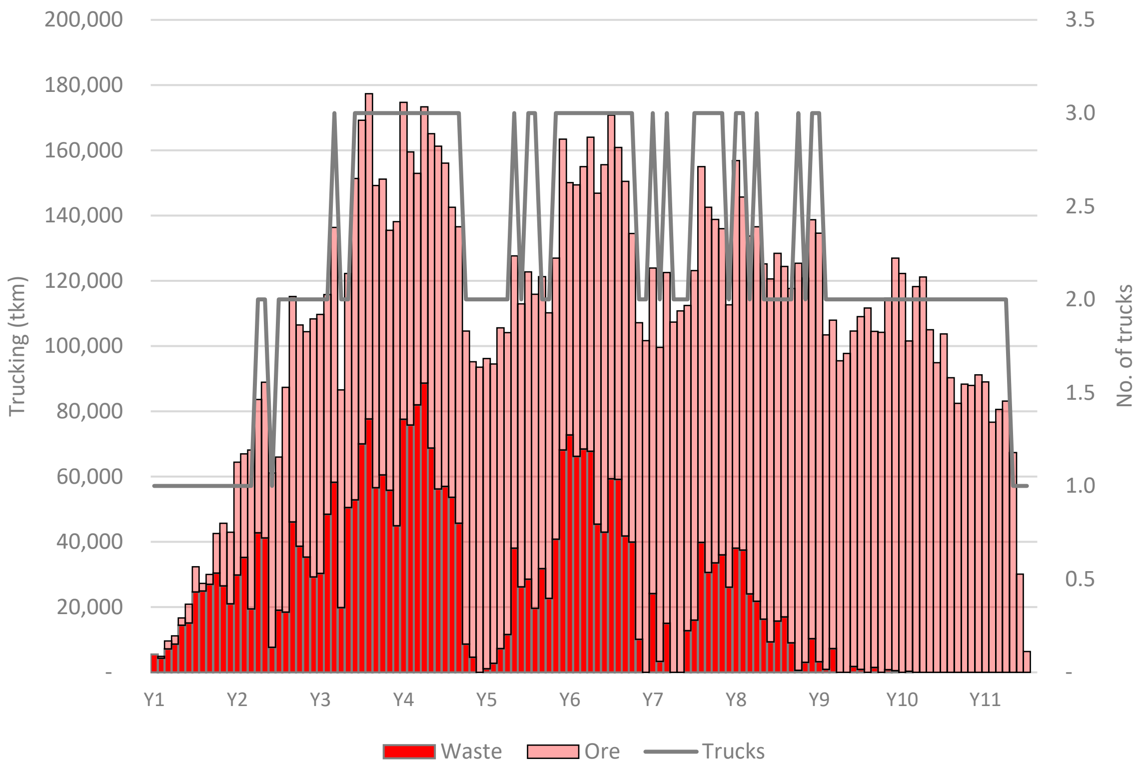


Figure 5-40 Haulage schedule

5.15 Grade Control Drilling

Grade control drilling will be critical for effective mine planning and for the delivery of forecast grades because the tin is distributed within the tourmaline unit and it is likely that visual grade control will be difficult. A grade control drilling program has been incorporated into the mine plan which involves drilling out the orebody on 15 m spacing. The drill design uses existing stockpiles where possible but also includes 1,310 m of dedicated diamond drill drive development to provide adequate FW drilling platforms. The program requirement is approximately 76,000 diamond drill metres over the life of mine.

An isometric view of the designed drill program is shown in Figure 5-41.

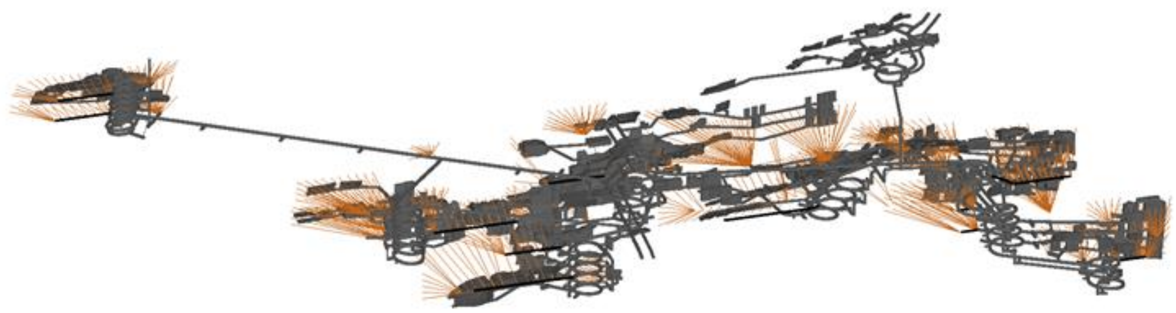


Figure 5-41 Designed grade control program (isometric view looking NE, diamond drill drives shown black)

5.16 Waste Material Balance

Waste rock will be generated from waste development. This waste will initially be trucked to surface for storage on the surface waste dump. Once backfilling of stopes commences, the waste will be trucked back underground into the voids and when possible directly from the blasted face to the stope void.

The total waste balance is positive, with approximately 1.8 Mt of waste rock being stockpiled at the end of mine life, less quantities used for contraction and other purposes. The waste balance for the mine plan is shown in Figure 5-42 below.

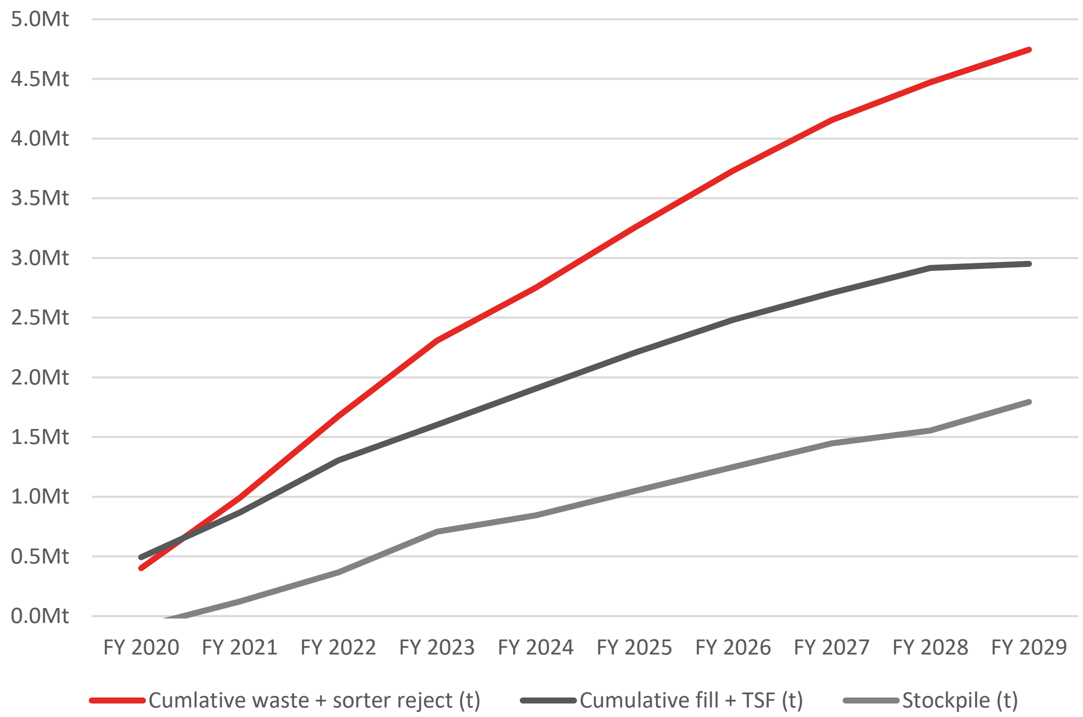


Figure 5-42 Mine waste balance

5.17 Mobile Fleet

The estimated major fleet equipment requirement is presented in Table 5-22.

Table 5-22 Major fleet estimate

Equipment	Equipment Type Example	Estimated Max. Qty
Jumbo	Atlas Copco M2D	3
Production drill	Atlas Copco Simba M7C	2
Development loader (7.0m^3)	CAT 2900	2
Production loader (7.0m^3)	CAT 2900	5
Truck	Atlas Copco 5010	3
Grader	CAT 12G	1
Integrated tool carrier	Volvo L120	2

Charge wagon	Normet Charmec	2
Shotcrete sprayer		1
Agitator truck		1

The key assumptions of the fleet number estimates are discussed below:

- Loader requirements include loaders to place backfill;
- Trucking requirements are based on the trucks dumping on a ROM pad or waste dump located near the portals.

5.18 Mine Ventilation

The ventilation plan is based on the planned mining fleet required by the mine schedule. The ventilation circuit was modelled using Ventsim® software to determine primary ventilation fan requirements assuming airflow demand on a staged basis for each decline according to the expected levels of activity.

5.18.1 Ventilation Circuit

The ventilation circuit comprises axial primary fans installed in walls at two exhaust locations, the Central ventilation portal and the Eastern ventilation portal, and a fresh air intake through the Western Zone fresh air raise to surface as shown in Figure 5-43.

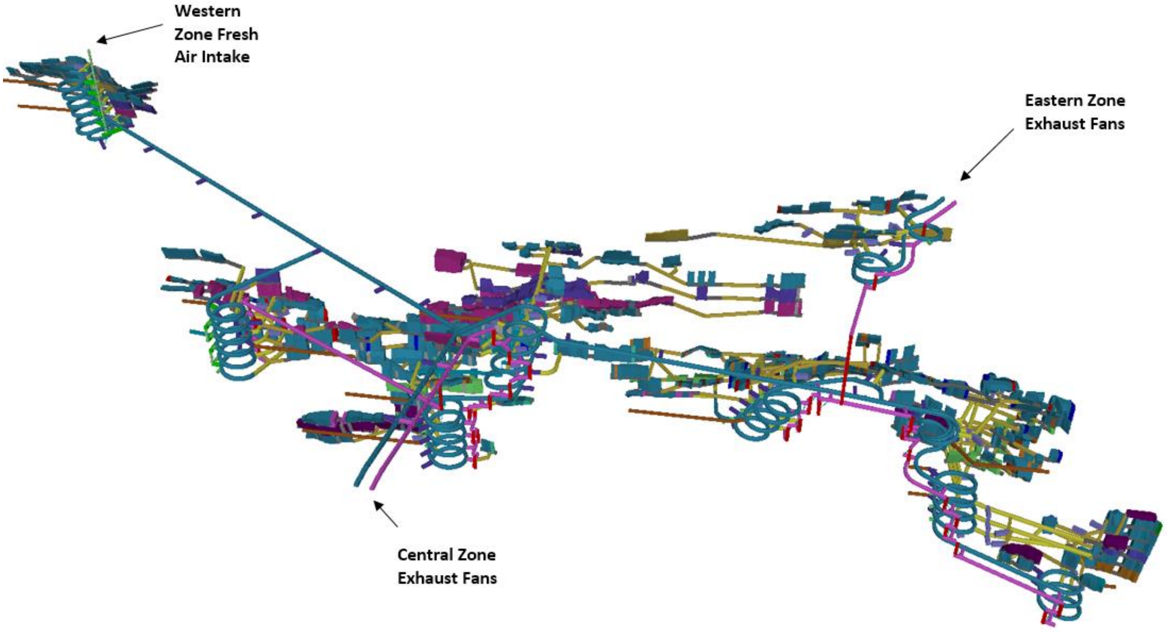


Figure 5-43 Ventilation circuit

Fresh air enters through the Eastern and Central access portals and travels into the mine through the declines. Fresh air for the Western Zone enters through the fresh air raise to surface and travels out of the area through the Western Zone access to join the ventilation circuit in the rest of the mine.

Fresh air will be transferred to working areas and faces through flexible ventilation ducting by auxiliary fans mounted in the declines. Return air will be exhausted through the return air drive and raise systems through the primary fans located in the Central and Eastern ventilation portals as shown in Figure 5-44.

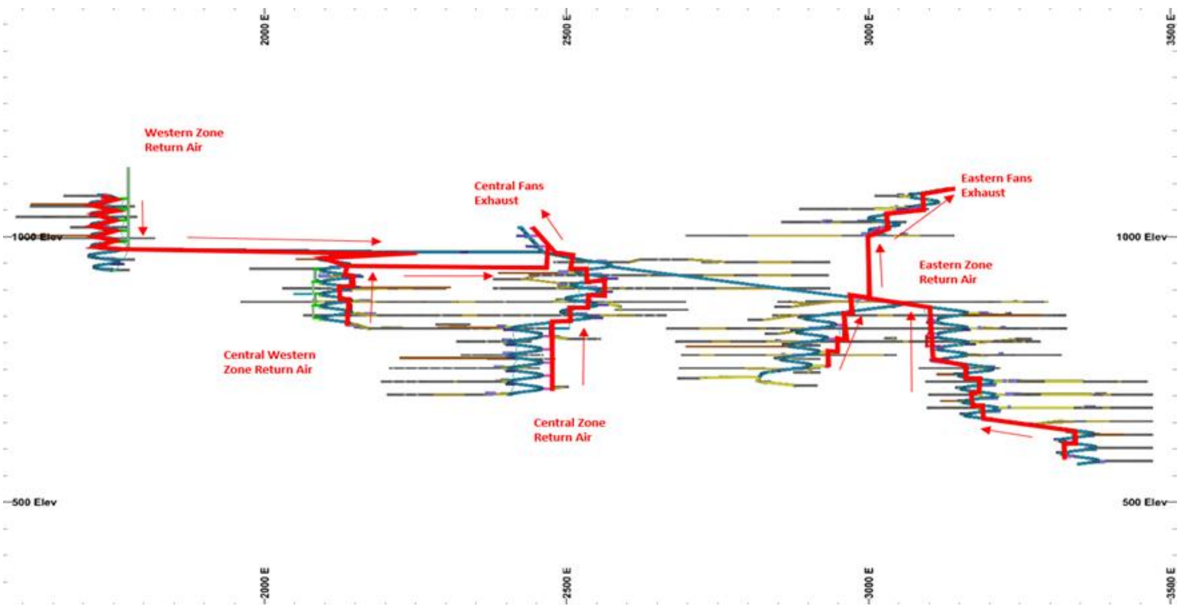


Figure 5-44 Achmmach mine plan return air route (red)

5.18.2 Ventilation Requirements

Ventilation volume requirements are based on estimates for fleet requirements in each zone using changing production profiles over the life of mine. A maximum air velocity of 6-8 m/s was assumed in travelways. Maximum ventilation requirements for the mine were determined using the West Australian mining regulations requirement of 0.05 m^3/s of airflow per rated engine kW power. The engine power ratings and maximum fleet numbers provided in Table 5-23 determine the maximum airflow requirement over life of mine for fan selection.

Table 5-23 Calculated minimum airflow requirements

Equipment Description	Make & Model	Engine Power (kW)	Ventilation per item (m^3/s)	Fleet size	Total Flow (m^3/s)
Truck	Atlas Copco MT5010	485	24.3	3	73
Loader-capital development/ loading	CAT R2900G	321	16.1	2	32
Loader-operating development / stope production / backfill	CAT R2900G	321	16.1	5	80
Agitator		164	8.2	1	8
Sprayer	Spraymec	96	4.8	1	5
Charge-up machine	Charmec	110	5.5	2	11
Integrated tool carrier	Volvo L120	107	5.4	2	11
Grader	CAT 12H	123	6.2	1	6
Underground magazine			15.0	1	15
Underground workshop			15.0	1	15
Leakage (20%)					41
				Total	**297**

Staged ventilation requirements were generated for each mining area based on the activity being undertaken. A minimum of 40 m^3/s was assumed for areas in which mining is taking place (representing the minimum required for a loader and a truck) and 15 m^3/s for mined-out areas with no mining activity (representing 0.5 m^3/s airflow for personnel access), with airflows increased as required based on the production requirements from that area. The staged airflow requirement estimates were used to generate air power estimates for the stages throughout the life of the mine for costing purposes. The stages are described in Table 5-24 and the results of the analysis summarized in Table 5-25

Table 5-24 Life of mine ventilation stages

Stage	Stage Start	Stage Finish	Stage Description
Stage 0	Month 1	Month 4	secondary ventilation from portals
Stage 1	Month 5	Month 22	central decline and upper eastern decline only-not connected
Stage 2	Month 23	Month 39	central decline only (upper eastern complete)
Stage 3	Month 40	Month 48	eastern and central (east vent shaft broken through)
Stage 4	Month 49	Month 60	eastern and central & wz (wz far broken through)
Stage 5	Month 61	Month 68	lower eastern and central, wz stoping
Stage 6	Month 69	Month 84	lower eastern and central stoping
Stage 7	Month 85	Month 127	lower eastern and central west stoping

Table 5-25 Air power requirement estimates over life of mine

Stage	Central Fan Power (kW)	Eastern Fan Power (kW)
Stage 0	0	0
Stage 1	81	16
Stage 2	150	0
Stage 3	230	211
Stage 4	77	488
Stage 5	152	494
Stage 6	163	528
Stage 7	163	528

5.18.3 Primary Fan Selection

A fan manufacturer (ECE-Cogemacoustic) was engaged to provide a quote for a primary fan system capable of generating the required airflows within the ventilation network. The manufacturer provided a detailed quote recommending two x 355 kW, 2.5 m diameter variable speed axial fans to be mounted in parallel in each ventilation portal as shown in Figure 5-45.

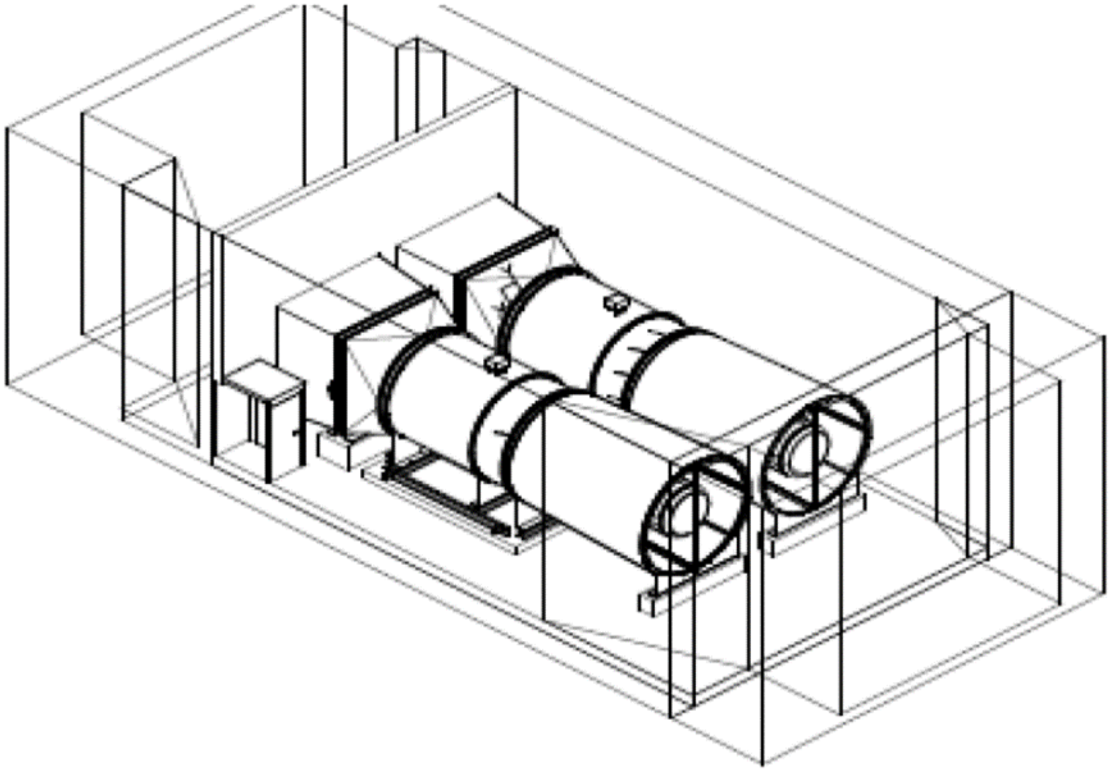


Figure 5-45 Proposed primary fan layout

The fan curves provided by ECE-Cogemacoustic are shown in Figure 5-46 (note that the curves represent a single fan, not the twin system).

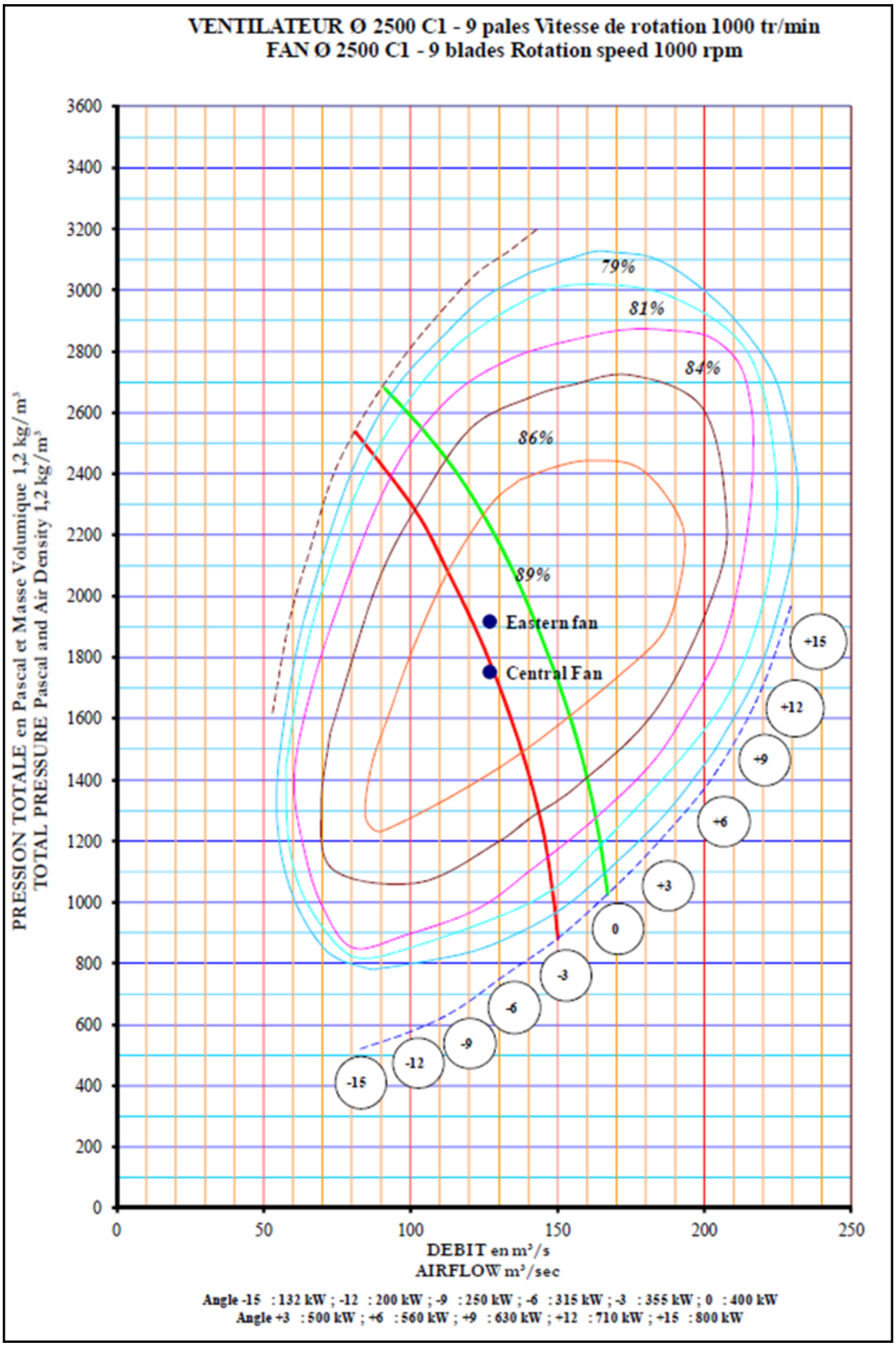


Figure 5-46 Fan curves for proposed primary ventilation system

5.19 Mine Services

5.19.1 Dewatering Infrastructure

The underground dewatering infrastructure schedule was designed according to the mine plan and on the basis that the mining contractor will provide secondary pumps.

Three-stage helical rotor pumps (Mono 103 – type) have been selected to be used for the primary pumping network. The dewatering design is based on a maximum inflow for the whole mine of 14 L/s, in which is the capacity a single Mono 103. All other pumps are required for stage-pumping to the main pump station (with a second Mono 103 installed for redundancy and to allow maintenance on the primary pump). The selected pipe sizing is sufficient to allow both pumps to run simultaneously to cope with short term inrush situations.

The proposed locations of the primary pumps are illustrated in Figure 5-47.

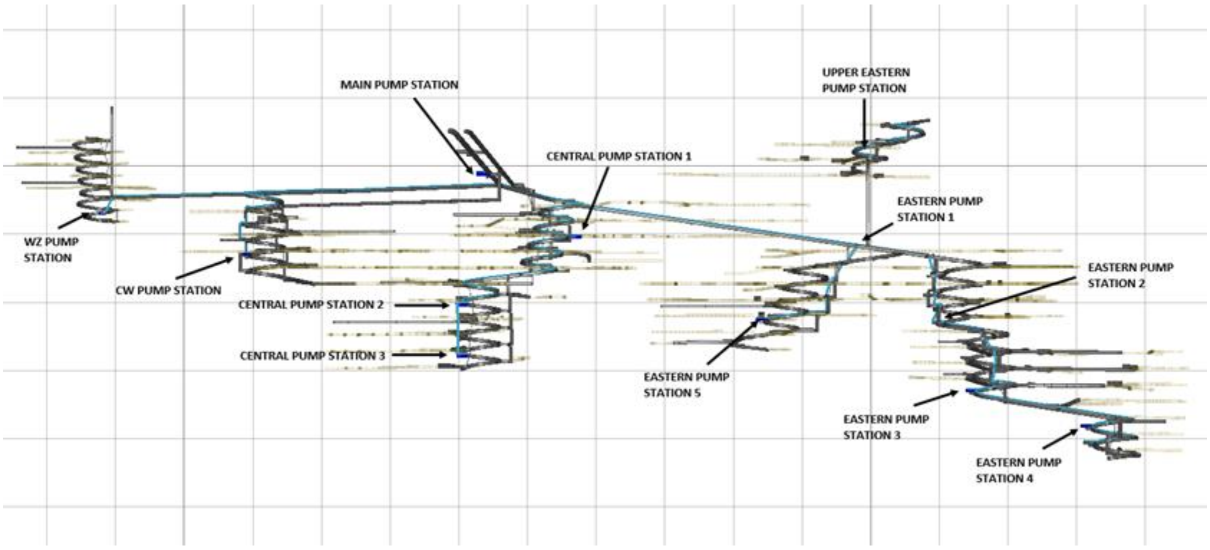


Figure 5-47 Primary pumping system pump locations

5.19.2 Power Infrastructure

Electrical power will be supplied as 11 kV feed to each portal and reticulated through the return air system (wherever possible) or the main decline to 11 kV/1 kV step-down transformers in working areas. All low voltage equipment and reticulation (including distribution boards and starter boxes) will be provided by the contractor and included in the provided rates. Sub-stations will be moved when work within an area is completed to minimize capital costs (e.g. sub-station 1A in the Upper Eastern area is moved to the Central Western area to become sub-station 1B). The final sub-station plan is presented in Figure 5-48.

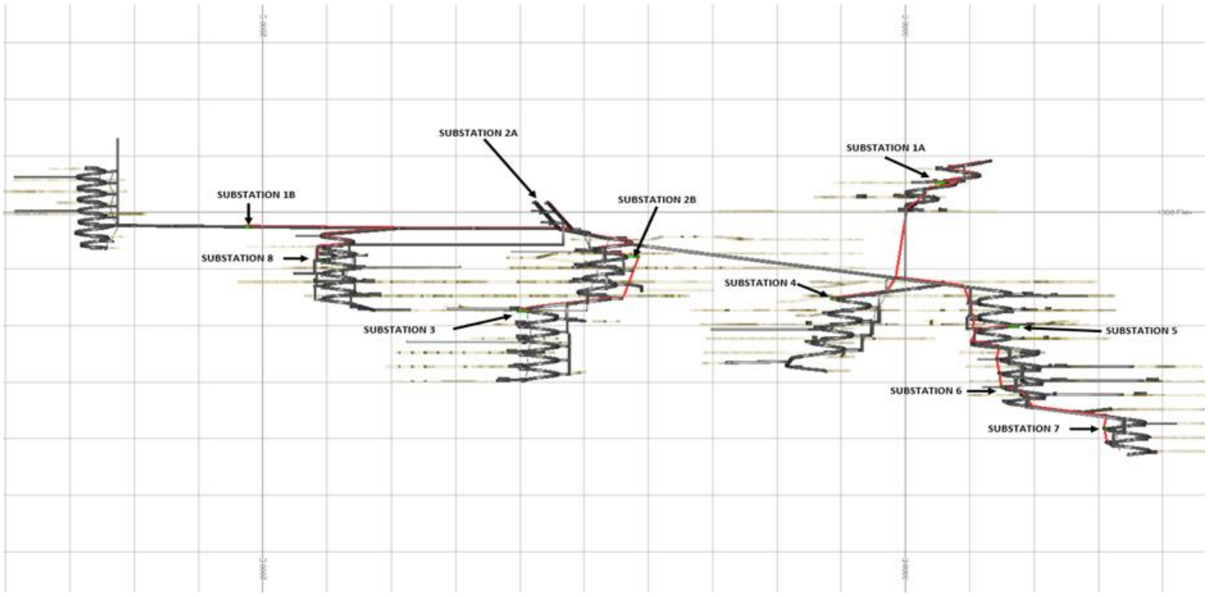


Figure 5-48 Electrical sub-station location plan

5.19.3 Compressed Air

Two 250kW Atlas Copco GA 160 compressors delivering 26m^3/min at 8 bar are proposed the service the underground workings. They will be located at the eastern and central portals.

Reticulation of compressed air is designed with 110 mm polyethylene pipe for the main service corridor and 64 mm polyethylene pipe for reticulating compressed air inside the levels.

It is proposed to use an ITH drill for long hole raising. This piece of equipment consumes a large volume of high pressure compressed air and will be supplied with its own portable compressor.

5.19.4 Secondary Ventilation

Working levels will be ventilated using auxiliary fans pushing fresh air from the primary ventilation circuit (decline) through flexible ventilation ducting.

The secondary fan requirements are based on the number of concurrent mining areas in the schedule. Secondary fans were sized at 110 kW will supply sufficient airflow for the planned mining fleet in the stoping areas. Similarly, 220 kW fans are planned for capital development ventilation where there are longer distances to ventilate and the mining fleet is larger. The fan schedules are summarized in Figure 5-49 and Figure 5-50.

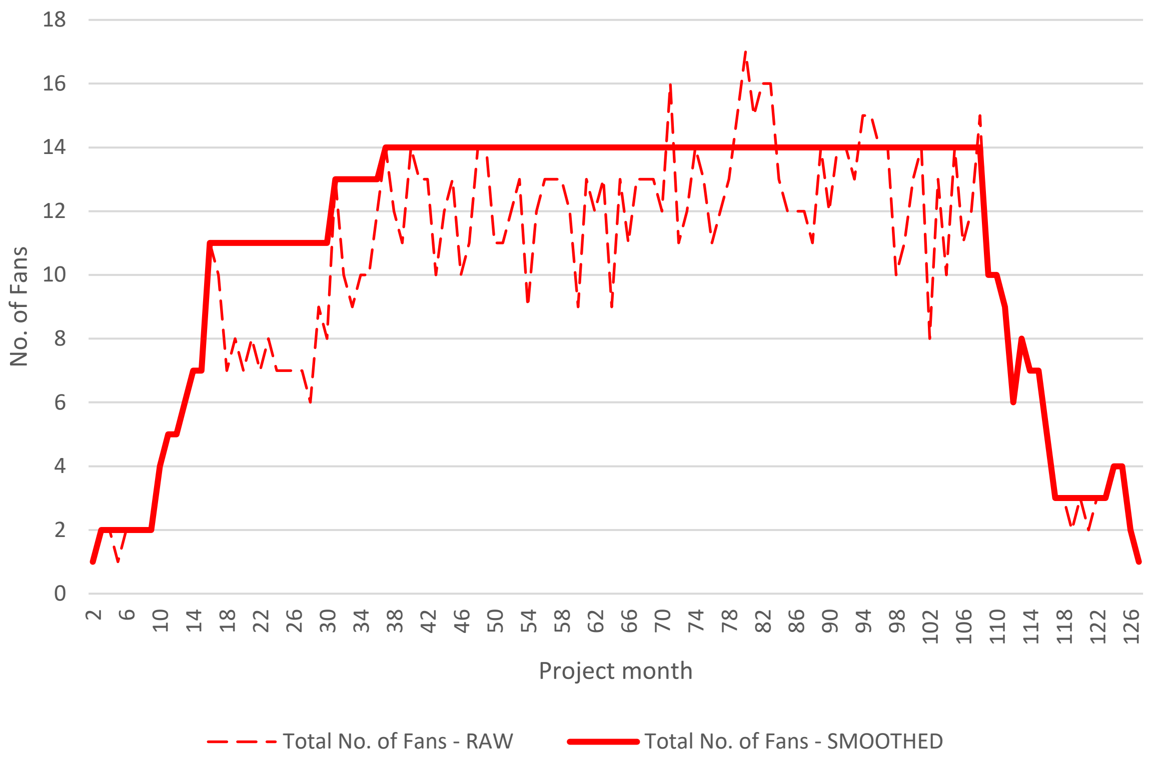


Figure 5-49 Operating auxiliary fan schedule (110 kW)

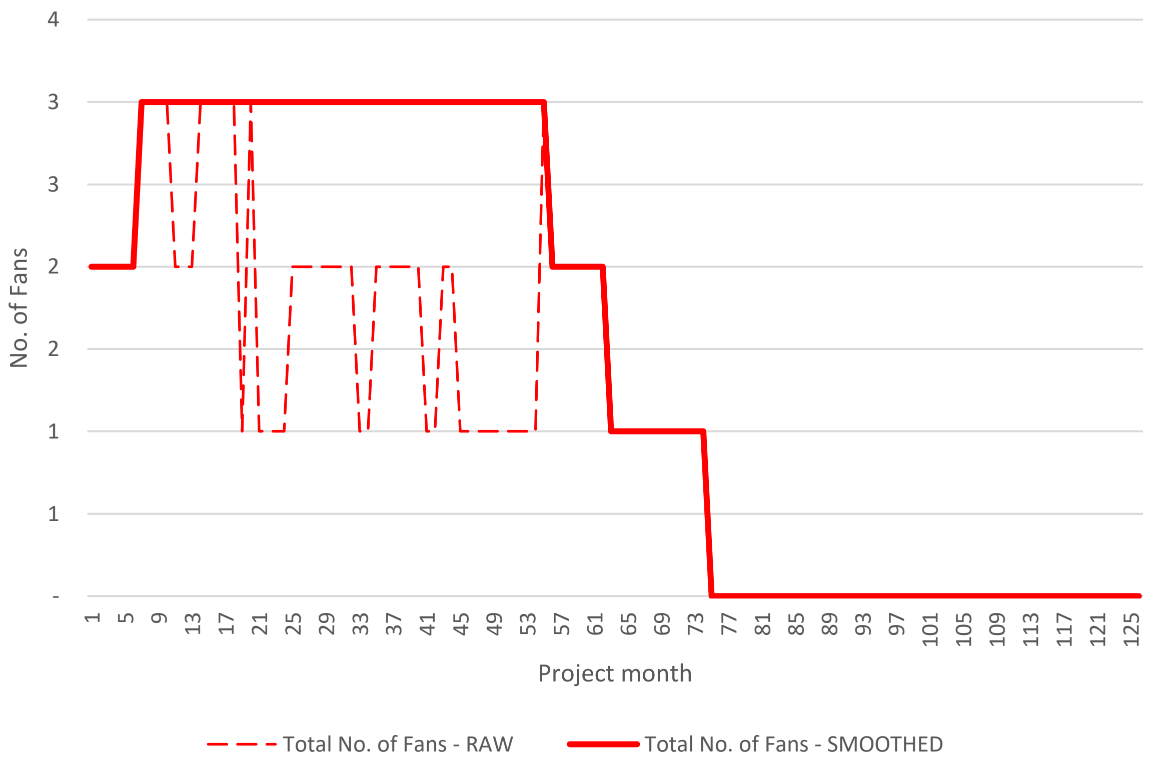


Figure 5-50 Capital auxiliary fan schedule (220 kW)

5.19.5 Communications

Underground communication will be transmitted through a leaky feeder UHF radio system. The communication network will be reticulated through the mine during development. To maintain the quality of communication, a series of communications huts and in-line boosters will be installed as the underground network progresses. The leaky feeder system will connect to the surface communication infrastructure near the portal and main electrical infrastructure.

5.19.6 Mine Water

Mine water will be provided from a header tank located on the surface above each of the portal locations. A 100,000 L tank in each location will meet the water demands of the operation.

During the initial stages of mining, water may need to be pumped from the tanks to provide adequate pressure for mining equipment to operate. If required and the water quality is sufficient, a portion of the dewatered mine water may be filtered and fed back into the mine water tanks. This will reduce the overall long term water demand from external sources.

Water will be reticulated through 110 mm polyethylene pipe for the main service corridor and 64mm polyethylene pipe for reticulating inside the levels.

5.20 Power Requirements

The estimated energy usage for the underground mine has been determined based on installed power and calculated utilization. The average power consumption is estimated to be 1,300 MWh per month.

The maximum estimated energy usage of 1,886 MWh occurs in Month 61. The breakdown of power draw in kW by equipment type for Month 61 is summarized Table 5-26.

Table 5-26 Month 61 average power draw breakdown at peak mine energy usage point

Equipment	Power Draw (kW)
Central primary ventilation fans	465
Eastern primary ventilation fans	360
Air compressors	102
Twin 110 kW stage secondary fans	267
Twin 55 kW stage secondary fans	624
Helical rotor primary pumps	79
Centrifugal secondary pumps	40
Refuge chambers	60
Underground lighting	60
Surface infrastructure	24
Mobile equipment	475
Total	2,556

The estimated peak mine power draw over the life of mining operations is presented graphically in Figure 5-51.

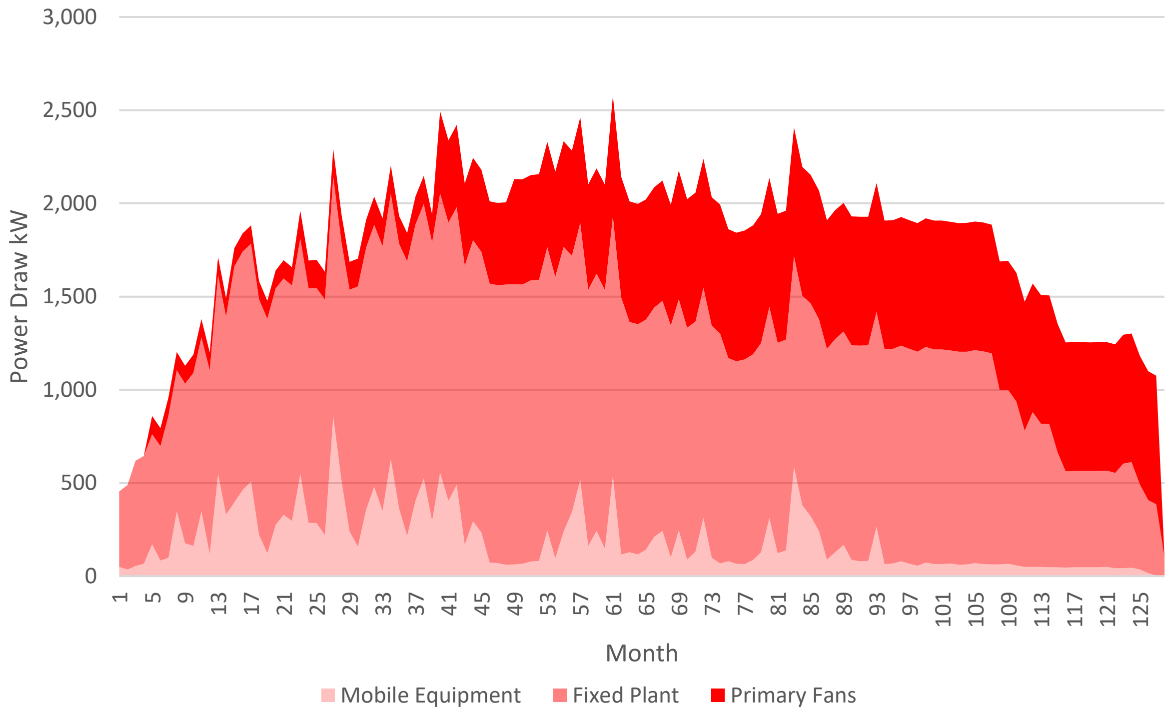


Figure 5-51 Peak power draw estimate over life of mine in kW

5.21 Emergency Egress

The emergency egress to the surface in the Eastern, Central and Western zone is provided through internal horizontal development and vertical raises 1.1 m in diameter. Each raise will be equipped with a caged ladderway.

The escapeways are connected to the surface through existing planned development as outlined in Section 5.7.3 and exit via the following locations:

- eastern portal – 1085 mRL
- central portal- 1015 mRL
- western fresh air raise – 1130 mRL

5.22 Ore Reserves Statement

The Achmmach ore reserve is presented below in Table 5-27. The reserve is based on a 0.55% Sn as a cut off for design.

Table 5-27 Achmmach ore reserve

Zone	Proved			Probable			Total		
	Ore (kt)	% Sn	Tin Metal (t)	Ore (kt)	% Sn	Tin Metal (t)	Ore (kt)	% Sn	Tin Metal (t)
Meknès Trend	1,100	0.99	11,000	5,600	0.78	44,000	6,700	0.82	55,000
Sidi Addi Trend	-	-	-	300	0.86	3,000	300	0.86	3,000
Total	**1,100**	**0.99**	**11,000**	**5,900**	**0.79**	**47,000**	**7,000**	**0.82**	**58,000**

6 METALLURGICAL PROCESSING

6.1 Introduction

The economic mineralisation in the Achmmach deposit is cassiterite SnO_2 (95%) with very minor amounts of stannite. Cassiterite is the primary ore of tin and a range of extraction methods have been developed that are suited to its characteristics. The metallurgical characteristics of cassiterite are:

- Being an oxide, it is very friable and excessive grinding will create very fine particles (slimes) which are more difficult to concentrate than coarse particles.
- It has an specific gravity of approximately 7 which makes it suited to gravity separation techniques.
- It is generally not as amenable to flotation as other base metals. Flotation tends not to be as selective and is only effective for a small particle size range thus producing a lower grade concentrate (<30% Sn).

The result of these characteristics is that cassiterite requires a comparatively more complex process flow sheet than many other base metals (eg Cu, Ni, Zn, Pb). It also requires considerable testwork to determine the response to the different methods and thus the most appropriate combination of extraction methods as well as the optimum particle size range at which to operate each method. The objective is to liberate the cassiterite at a coarse particle size with minimal undersize which is suited to the lower cost and effective gravity methods such as spirals and shaking tables, then apply the more sophisticated gravity methods such as centrifuges.

Methods such as magnetic separation are employed to remove iron as magnetite and other gangue.

Pre-concentration after primary crushing using sorting technology is effective with cassiterite as it can operate on multiple mineral characteristics (eg density by XRT). Pre-concentration increases feed grade, reduces plant size and reduces feed grade variability thus making the downstream process more effective.

6.2 Metallurgical Testwork

6.2.1 Bulk Samples

The resource diamond drilling programmes have produced NQ (48mm) and HQ (63mm) core. The core has been halved with one half set aside for general reserve and metallurgical analysis. The other half of the core is normally interval cut at 1 m lengths, crushed to 75% passing 2 mm, split and assayed for resource definition.

The balance of the crushed half-core from resource drilling programmes formed the basis of the bulk Meknès composite (BMC) and Eastern Zone shallow (EZS) composite. From the original 6 tonnes of BMC material, 3 tonnes were split out for the 2014 study and the remaining 3 tonnes used in the 2015 study.

All drill holes sampled for the BMC are located within the mineralised envelope. Figure 6-1 depicts the spatial representativeness of the selected BMC and EZS in relation to the wire-frame model.

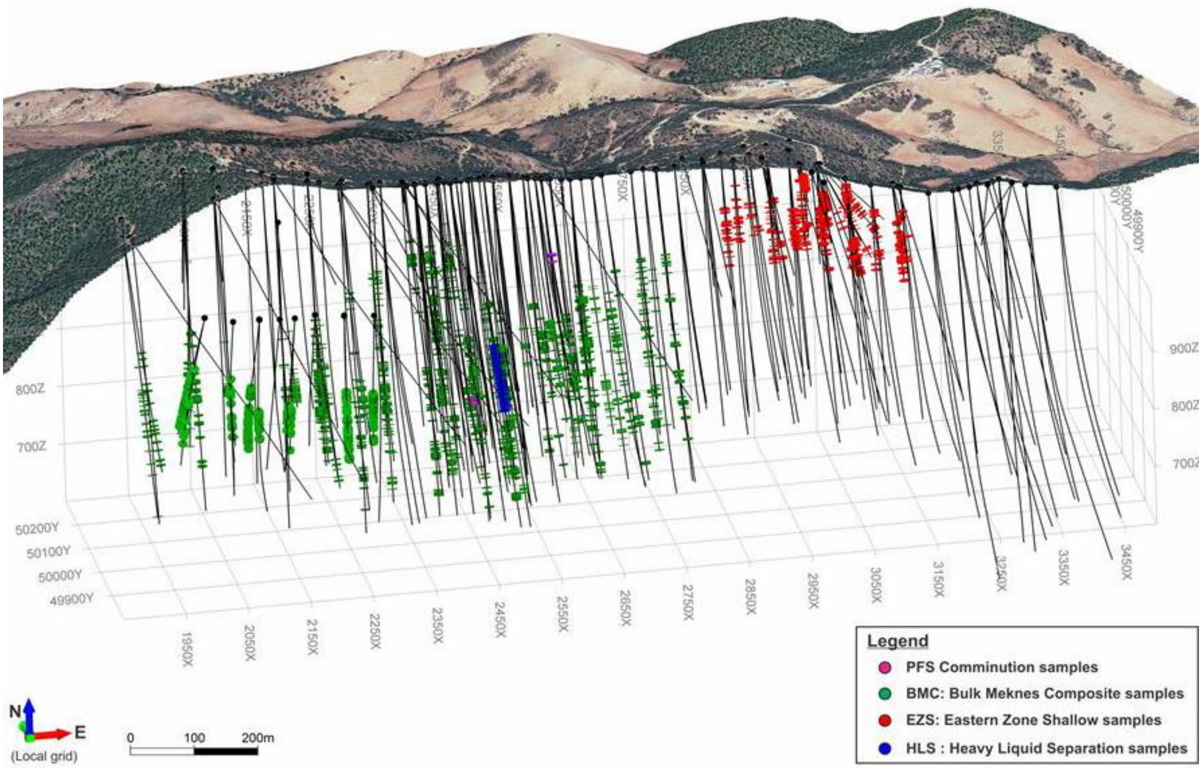


Figure 6-1 Meknes trend metallurgical sampling

The two main testwork composites are considered to be adequately representative of the orebody. mineralogy.

6.2.2 Testwork Program

Collectively, a new power efficient comminution circuit including the new high efficiency ore sorting systems, evaluation of a coarser grind size, the impact of these on the flotation circuit response and the approach to dressing of flotation concentrate necessitated further test work. The objective of this test work was to drive down comminution risk and power draw and confirm the gravity and flotation grade recovery response and its cost structure in the context of the new comminution system.

The 2018 test program utilised four large samples of drill core (~500kg each) taken from the main ore zones along strike. Each sample was taken from the drill core intervals from inside the ore blocks and includes a mining dilution envelope. The samples combine material from 119 drill holes, and over 1,600 one metre intervals of core. The core size varied between NQ and HQ and was mainly quarter core. The details are summarised in Table 6-1 below.

Table 6-1 Summary of 2018 testwork samples

Zone	Sample #	Number of holes	Metres of core
Eastern	1+2	33	380
Middle 1	3+4	29	425
Western+Middle	5+6	36	424
Middle 2	7+8	22	376

These samples are each representative of the four ore zones along the strike of the orebody. Western zone is only 300 kt of the ~7500 kt in the reserve and had to be topped up with adjacent central zone material to have sufficient sample for the test work. The grade distribution of the three key samples is shown in Figure 6-2 below. This is consistent with the grades achieved in ore sorting tests, where with 40-45% mass rejected the rejects grade ranged between 0.1% Sn and 0.2% Sn.

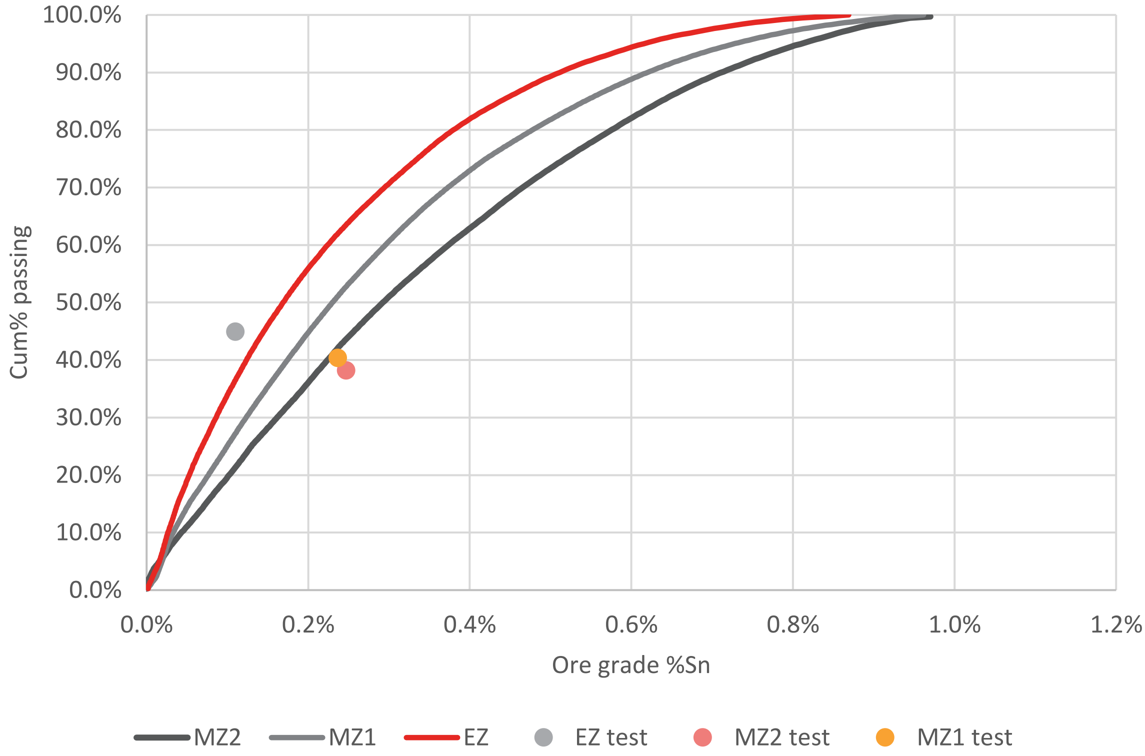


Figure 6-2 Grade distribution

The 2018 test work investigated a new front end crushing and grinding regime with an ore sorting stage incorporated between conventional crushing and high pressure grinding rolls (HPGR). The grindability testing on the HPGR product defined for each of the ore zones the impact of these changes on the primary closed circuit ball mill feeding the gravity and flotation recovery circuits.

The four samples were crushed to a P_{80} of 32 mm, then screened at 10 mm aperture. A collection of lumps of core split into three grade ranges was analysed on a Steinert KSS ore sorter to develop an algorithm for later bulk testing. A 50 kg sample of the >8mm material from the dominant central ore zone was then tested and sampled for assay to define how the separation algorithm functioned. Based on these results the algorithm was adjusted and the four 500kg samples were separately tested and the products sampled and assayed.

A sample of <8 mm fines was tested by heavy liquid separation to provide preliminary data on jig processing of the fines to enable further reduction in mill tonnage throughput and further upgrading.

The 8 mm to 32 mm ore sorter reject material was partly re-crushed to create an adequate amount of <8mm fines and then test crushed in a 1.0 m diameter Koeppern HPGR. This test provided design data at three different pressure set-points.

The four samples of accepted ore sorter product were then recombined with the respective <8mm ore fines samples and crushed in a 1.0 m diameter Koeppern HPGR at the best previous test pressure.

The accepts plus fines from the four ore zones crushed in the HPGR were then split into samples for rod and ball mill Bond Work Index testing, quantitative QemScan mineralogy, jar mill testing for the regrind duty, packed bed testing by Metso in York to provide additional HPGR design data, and flotation testing.

The other driver for the additional grinding testwork was the impact on grindability of rejecting 40% of the low-grade ore mass in the ore sorting process. Observation of drill core from the ore zones has indicated that the highly tourmalinised low grade halo around the main quartz veins bearing the high grade cassiterite could be the harder material.

The flotation testing was conducted on a 60 kg composite of the upgraded ore after recovery of gravity amenable cassiterite, where the gravity response enabled comparison of grade recovery performance for both 150 µm and 106 µm P_{80} grind sizes. Flotation was not used to assess this primary grind size change because the flotation is only utilised on <38 µm material, however the <8 µm slime tail mass flow created downstream of this test was used as part of the evaluation.

A second program examined a range of collector additions for a range of collector types and blends. The best results of all these tests were used in a locked cycle test using the flotation regime developed by ALS & Toyota Tsusho Co (TTC). This work collectively also defined the impact of the ore sorter upgrade and the HPGR crushing.

Flotation concentrate will be tested on several wet high intensity magnetic separation (WHIMS) devices to define the potential for reducing iron levels and achieving sufficient upgrading to minimise the Falcon dressing process for flotation concentrate. This work will form a part of the final testing prior to the project FEED phase.

The test work program is shown schematically in Figure 6-3.

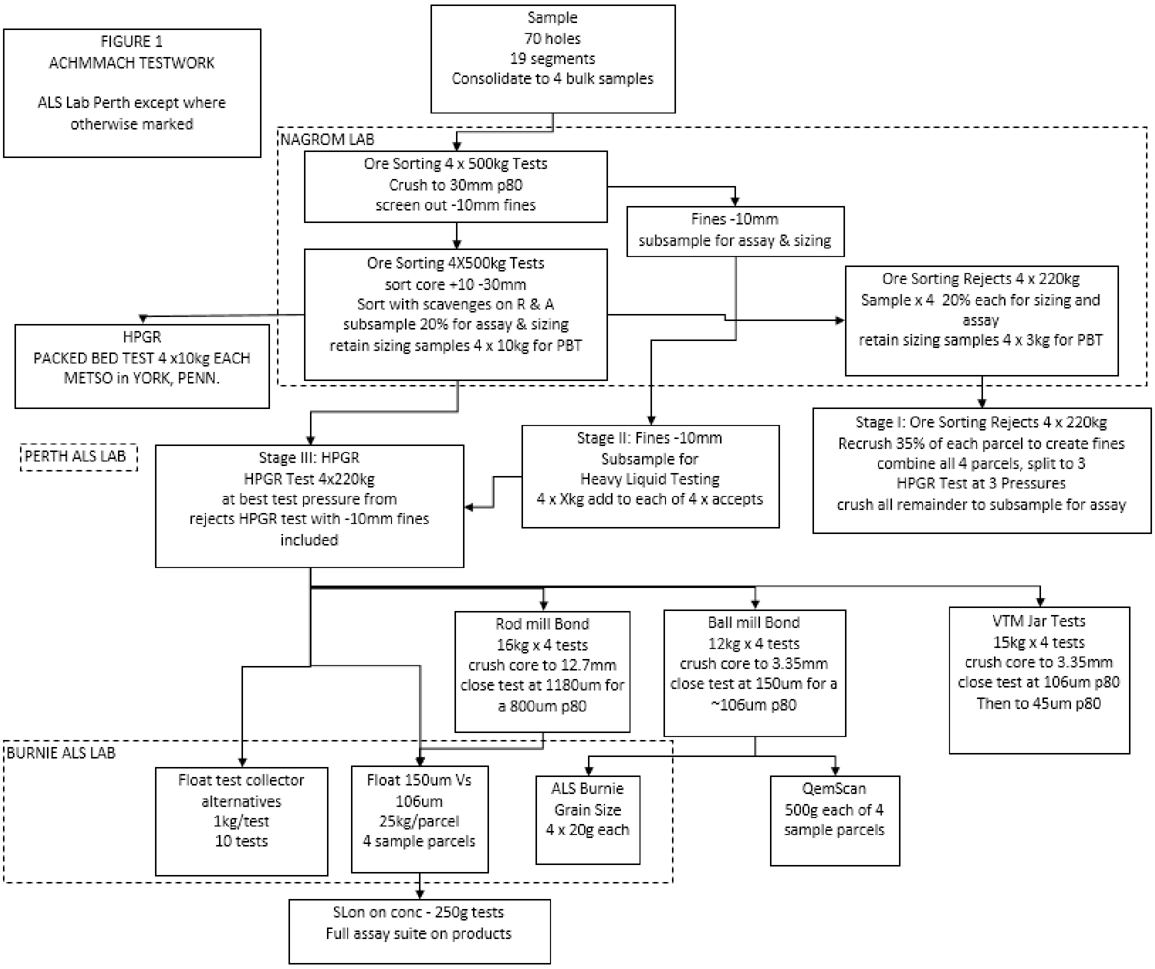


Figure 6-3 Test work program flow diagram

6.2.3 Work Index

The results from previous crushing work index tests conducted by Atlas Tin have been used in the study. The bond crushing work index test and UCS data in Table 6-2 have very large scatter, and both indicate very high values. The peak values are being used in design, so crushing plant equipment selection and plant design are not considered to be especially vulnerable to variability.

Table 6-2 Summary of work index test results

Parameter		No. samples	Unit	Result
SG(by wax coat & immersion)	average	6		2.81
	maximum	-		2.90
SG (from CWi tests)		-		2.95
Unconfined compressive strength	average	13	MPa	115.2
	maximum	-	MPa	204.5
Strength class		Strong to very strong		
Crushing work index, CWi	average	11		9.90
	maximum	-		22.40
Abrasion index, Ai	average	2	0	0.792
	maximum			0.895
Rod mill work index, BRWi	average	2	0	30.1
	maximum			30.2
Ball mill work index, BBWi	average	2	0	24.6
	maximum			26.3

Additionally, the current two stage crushing circuit is targeting a product size of 32 mm, thus avoiding the more strenuous duty in the previous flowsheet where 12 mm to 16 mm product size was required for rod milling. This size is selected primarily for the operation of the downstream HPGR. Many of the HPGR units available at the throughput required are fitted with 1.0 m diameter rolls, which requires a feed size around 30 mm to 35 mm to achieve the nip angle for effective intake of feed into the roll gap.

6.2.4 Ore Sorting

The ore sorting test work was successful at the 32 mm crush size after removal by screening of the <8 mm fraction. A rejection of >40% of >8 mm ore to a grade of generally <0.2% Sn was achieved, and the variation from sample to sample was minor. In a full-scale plant, the proportion of fines will be almost double the amount from this test, so the degree of upgrade will be reduced.

Table 6-3 Ore sorting test results

		ore %Sn core assays	ore %Sn test assay	rejects %Sn test	accepts %Sn test	SnD% to accepts test	rejects mass sorter
1+2	EZ	0.87	0.92	0.11%	1.60%	94.7%	51.4%
3+4	MZ1	0.96	1.10	0.24%	1.69%	91.3%	45.0%
5+6	MZ1+WZ	0.66	0.70	0.15%	1.32%	88.6%	58.9%
7+8	MZ2	0.97	0.93	0.25%	1.36%	89.9%	43.6%

Using these results and projecting performance with 30% of <8 mm fines bypassing the ore sorter, a plant feed grade of 1.18% Sn is anticipated from a feed grade of 0.90% Sn with a 40% rejection in the ore sorter. Although the ore sorter technology can deal with particles smaller than 8mm in size, the capacity of the ore sorter is dramatically reduced.

A feature of the ore sorting tests was that sulphide minerals were also identified by the XRT system as high density mineral phases and so sorted to accepts.

The ore sorting testwork is presented in detail in Appendix 6A, along with further details of the selection of samples for optimum representivity.

6.2.5 High Pressure Grinding Rolls

The four variability samples were processed through a Koeppern test high pressure grinding rolls (HPGR) unit at ALS Perth, the specification of which is shown in Table 6-4. The initial test was conducted on each ~250kg sample of the ore sorter reject material, which had been partly re-crushed to create a normal size distribution. This was tested over four pressure settings to determine the design m-dot value to enable scale up of the test to a full scale machine.

Table 6-4 HPGR pilot plant specification

Item	Specification
Roller diameter	1,000 mm
Roller width	250 mm
Press drive	Planetary gear box
Feed system	Gravity
Wear surface	Hexadur WTII
Installed power	264 kW
Maximum pressing force	1,600 kN
Maximum specific pressing force	8,500 kN/m^2
Variable speed drive	Up to 21 rpm (1.1m/s)

The HPGR test results are shown in Table 6-5. These results are regarded as conservative due to the difficulties experienced in choke feeding the test unit. A degree of 'roll chatter' was observed in the test, due largely to the small quantity of the test sample. In a full scale operation this should be prevented through choke feeding.

Table 6-5 HPGR test results

Description	Test number	Sp pressing force (kN/m^2)	Sp throughput m- dot (ts/hm^3)	Net Sp. Energy (kWh/t)	Centre P_{80} (mm)	Centre P_{50} (mm)	Passing 12mm (%)
Evaluation of pressure 1st pass	ACH001	2,500	237.4	1.53	10.46	3.14	81.67
	ACH002	3,500	231.3	2.02	8.36	2.34	86.27
	ACH003	4,500	220.5	2.57	7.66	1.9	87.93

The sorter accepts plus the <8 mm fines were crushed in the HPGR at the middle pressure setting to create fine feed for Bond Work Index testing and jar mill testing. The bulk of the crushed accepts plus fines sample was then used for the flotation test work which is discussed below. The ground sample from the Bond Work Index ball mill test was used for QemScan mineralogical analysis and the product of the rod mill Bond Work Index test for cassiterite grain size analysis.

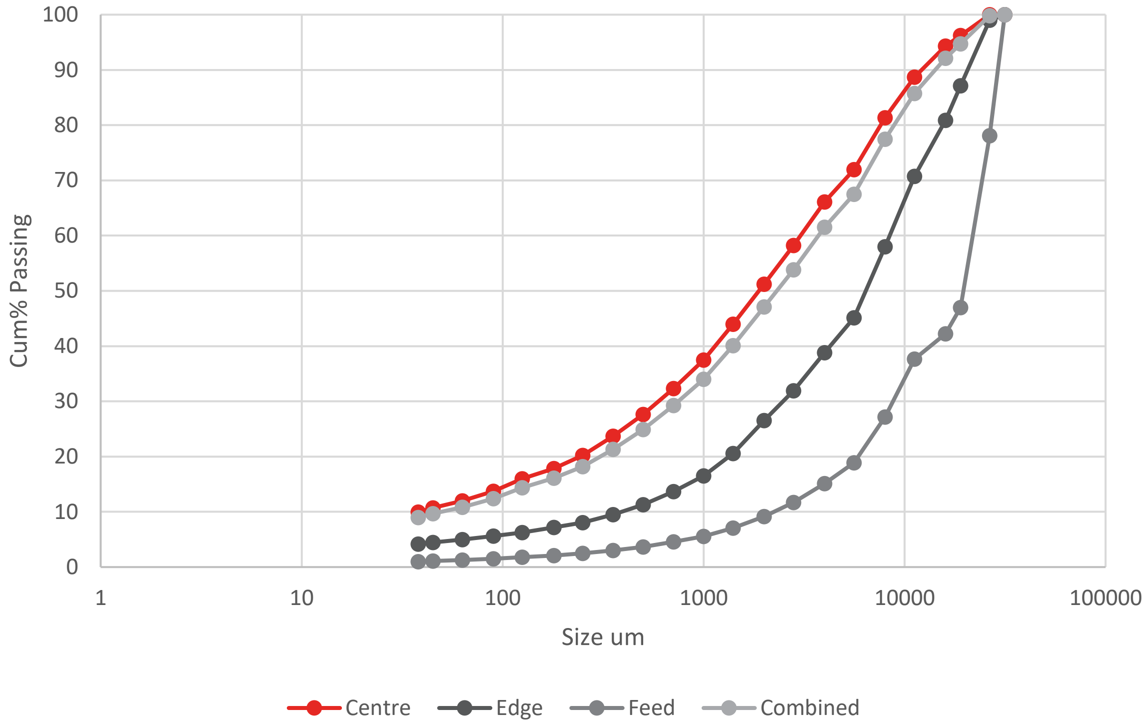


Figure 6-4 Particle size distribution ACH 003

The HPGR test work was successful in that nominally 32 mm feed was reduced to ~8 mm P_{80} at very low power draw. The reduction ratio is expected to improve given well controlled choke feeding at full scale, and potentially operating at slightly higher than test pressure for a small increase in power consumption. The power consumption of the HPGR was 1.5-2.0 kWhr/tonne ore.

The Bond Work Index testing of the HPGR crushed accepts plus fines showed that rod mill and ball mill work indices were reduced from previous values, probably due to the increased fines generated by the HPGR compared to conventional crushing. For some ores the high compression crushing is thought to contribute to this improvement in grindability due to micro-fracturing of the rock, however this is not readily established.

Table 6-6 Bond Work Index testing

Test	Previous values	New values	Closing size	Design Value
Crushing Wi	16-32	-	-	27
UCS	91-274	-	-	228
Rod Mill Wi	30-34	26-30	880µm	30
Ball Mill Wi	21-26	22-25	106µm	24
Abrasion Index	0.42-0.81	-	-	0.73

The jar milling test data indicates that:

- ore variability in terms of grindability is low
- that a stirred milling regrind stage would be power efficient
- that ore hardness for grinding remains reasonably consistent down to fine sizes.

These results add confidence to the selection of regrind mill power and also enables the selection of a stirred mill for the regrind duty.

Table 6-7 Jar mill predicted specific energy

Sample Id	F80 (µm)	Jar mill specific energy (kWh/mt) P_{80} 106 µm	Jar mill specific energy (kWh/mt) P_{80} 45 µm
Lot 1-2	628.4	7.55	14.03
Lot 3-4	618.2	8.29	14.49
Lot 5-6	614.8	6.59	12.18
Lot 7-8	563.4	7.77	13.94

Table 6-8 Jar mill test data interpretation

Test	~600 to 45µm	~600 to 106µm	106 to 45µm
Lot 1+2	14.03	7.55	6.48
Lot 3+4	14.49	8.28	6.21
Lot 5+6	12.18	6.59	5.59
Lot 7+8	13.94	7.77	6.17
Mean	13.66	7.54	6.11

The variability of results of the four ore types in comminution testing was very small. This is consistent with both geotechnical data, and with the geological model and its observation of relatively uniform tourmalinisation of the ore zones.

6.2.6 Gravity Processing

Sufficient gravity test work has been conducted over several study phases, including pilot scale testwork in WA and in South Africa, to fully define gravity separation response on spirals and tables.

The 2018 program was done in batch and used purely to:

- evaluate the optimum grind size
- evaluate the potential change in cassiterite deportment between gravity and tin flotation.

A 48 kg composite was formed from the four ore zones and processed in bulk to create sufficient material for the flotation test matrix and consequently to produce sufficient concentrate for the final wet high intensity magnetic separation testing. The gravity recoverable free tin was recovered first from the 38 µm to 212 µm fraction by three stages of tabling.

The composite was ground to nominal 106 µm (to create feed samples for flotation testing) with sufficient material ground to 150 µm for the grind size comparison by gravity separation on tables.

These samples were all then cyclone split at 38 µm to create the >38 µm material for gravity tests and the <38 µm for desliming at 8 µm. The 8 µm to 38 µm material was used for flotation testing.

The slime tail grade was very low, with significant upgrading of tin to the cyclone underflow at each stage in both the gravity and tin float. However, no gravity regrind step had been performed so this is considered an underestimate in terms of mass split.

The gravity testing was initially not a locked cycle test program for complete definition of the gravity response, as this has been established in many programs over many feasibility testing programs. It was undertaken to provide a comparison between the 150 µm and 106 µm grind sizes and to avoid the flotation sample being biased by a high free cassiterite population.

The results of this test work are shown below in Figure 6-5, which shows the response to gravity processing was almost identical for the 106 µm and 150 µm grind sizes. A lower concentrate grade was generated for the 150 µm grind, however the recoveries were the same. This concentrate in subsequent tests improved with regrind of the rougher and cleaner table tail. The recent mineralogy by QemScan indicates that the liberation is similar at these sizes, confirming the batch gravity outcome above and supporting the view that both significant primary grinding energy can be saved, and generating a lower mass of slimes.

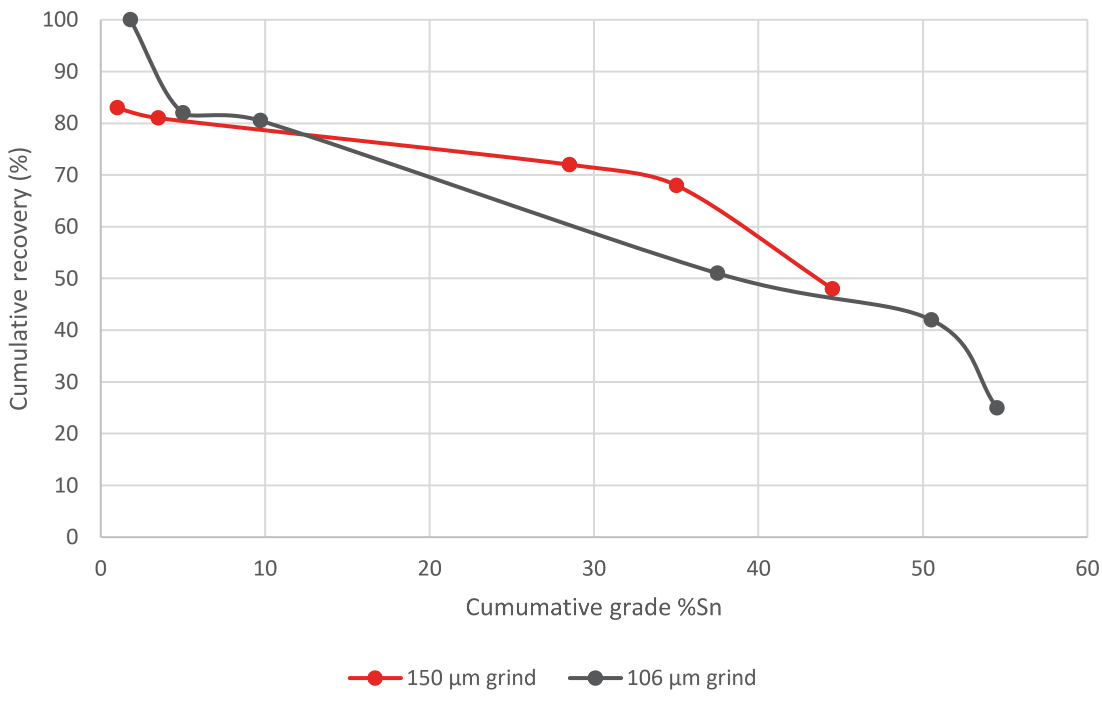


Figure 6-5 Grade recovery comparison for grind size

6.2.7 Low Intensity Magnetic Separation Circuit

Low intensity magnetic separation (LIMS) has been incorporated into gravity concentrate dressing and before sulphide flotation in preparation for cassiterite flotation feed.

No testwork was conducted on this step as the objective is simply to remove steel scrap, pyrrhotite and any magnetite that may be present. This is a standard application for a 1200 Gauss LIMS. With appropriate machine configuration and spray washing of magnetics the tin reporting to magnetics is negligible.

6.2.8 Flotation Circuit

The objectives of the flotation test program were to:

- assess the change from 106 µm p80 to a 150 µm P_{80} grind size on grade and recovery performance, including an analysis of the slime tail loss
- examine the potential for lower cost collectors and lower dosage rates
- confirm that the flotation regime performance in locked cycle in terms of grade recovery is consistent with previous locked cycle work.

Four different collectors were tested and found to be less effective than SPA-a styryl (2-phenylethyl) phosphonic acid which is also the preferred collector used at the Renison tin mine in Tasmania and historically in other tin flotation operations in UK and Canada. The lower cost OPS30 is a phosphate-based collector which demonstrated some selectivity. However, to achieve 88% rougher recovery, approximately 50% of the mass was floated compared with 36% for SPA. The Renison mine has successfully plant tested a blend of OPS30 and SPA and this approach is recommended for future testwork on the Achmmach ore.

A locked cycle flotation test has been completed on a composite of the four ore zone samples. This confirms the performance obtained in previous testing using the new feed preparation technique.

6.2.9 Falcon Dressing

The Falcon centrifugal gravity concentrator was analysed in previous testwork program to determine the applicability of these machines to the Achmmach mineralisation. This work included a large scale test on concentrates from a bulk flotation test. The outcome of this test was that:

- At a pilot scale the UF Falcon concentrator was able to upgrade 10% Sn concentrates to >40% Sn concentrate. However, to achieve this result a very complex and high capital cost circuit was required, which was not accounted for in previous work.
- The Falcon concentrator did not reject tourmaline effectively and the upgraded concentrates were high in iron, which incurs significant smelter penalties.

With the ability to generate high grade cassiterite flotation concentrates, and a high grade gravity concentrate from ore sorter accepts, it is anticipated that the WHIMS iron removal process will provide sufficient upgrade of tin in flotation concentrate due to the mass of iron bearing mineral removed.

A basic UF Falcon circuit similar to one successfully operated at the Renison mine has been adopted as the baseline dressing approach for flotation concentrate. This comprises three units in a rougher, cleaner and scavenger cleaner circuit. The rougher tail is recycled to the float circuit, and the scavenger tail is currently disposable to final tail. This may in practice also be recycled to flotation via the deslime circuit.

6.2.10 High Intensity Magnetic Separation Circuit

The Slon high intensity magnetic separator is an attractive innovation over the traditional carousel style WHIMS machines. It is lower cost and more efficient with none of the blockage problems that occur with carousel systems. Previous test work indicated high efficiency removal of iron in the region of 75%, presumably in sulphides and tourmaline, for a small tin loss. This test work will be conducted on low grade flotation concentrates but was not completed in time and so will be reported in an addendum to this study.

The study will utilise the slightly lower grade concentrate specifications generated in previous work for the purpose of defining smelter return on concentrates.

6.2.11 Concentrate Filtration

The quantity of concentrate produced from the dressed gravity and dressed flotation processes is small at 1.2 t/hr. The split is roughly 65/35; gravity to flotation concentrate. The gravity concentrate is relatively coarse particle size (>45 µm) and there are no slimes in the flotation concentrate having had <8 µm material removed.

Only very small quantities of concentrate are available from testwork. The concentrate will be dewatered in a plate and frame pressure filter which will readily deliver moisture levels in the range 7% to 11%. Consequently, no specific filtration testwork has been carried out by vendors.

6.2.12 Tails Thickening

Tailings thickening by compression in a paste thickener has been selected for the plant to maximise the return of process water to the process.

Atlas Tin has previously determined the yield stress characteristics of high density thickened tailings for the purpose of designing a paste backfill system (which is no longer planned). Yield stress characteristics are the main factor for specification of a 'high compression' paste thickener, which is a hybrid of high rate and compression. This has been selected because the density required of the tailings is approximately 70% to 72% solids.

The optimisation of water return is important in the context of a water storage facility filled by rain catchment. The higher than usual density remains pumpable by centrifugal pumps rather than requiring positive displacement pumps which is the case at densities greater than 72% solids. At this density the deposition forms beaches that do not segregate and so are free draining.

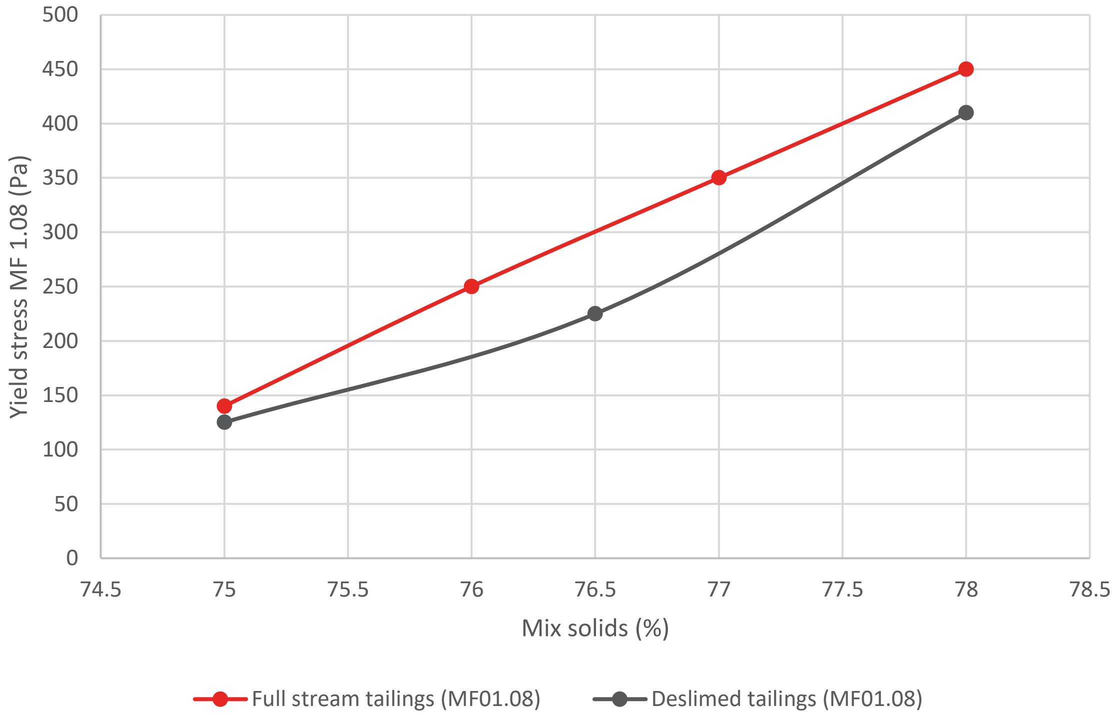


Figure 6-6 Tails yeild stress Vs density

In 2017 Atlas Tin carried out a series of definitive rheological tests on tailings from the TTC testwork program. This provides design data for both thickening and tailings pumping to the tailings facility.

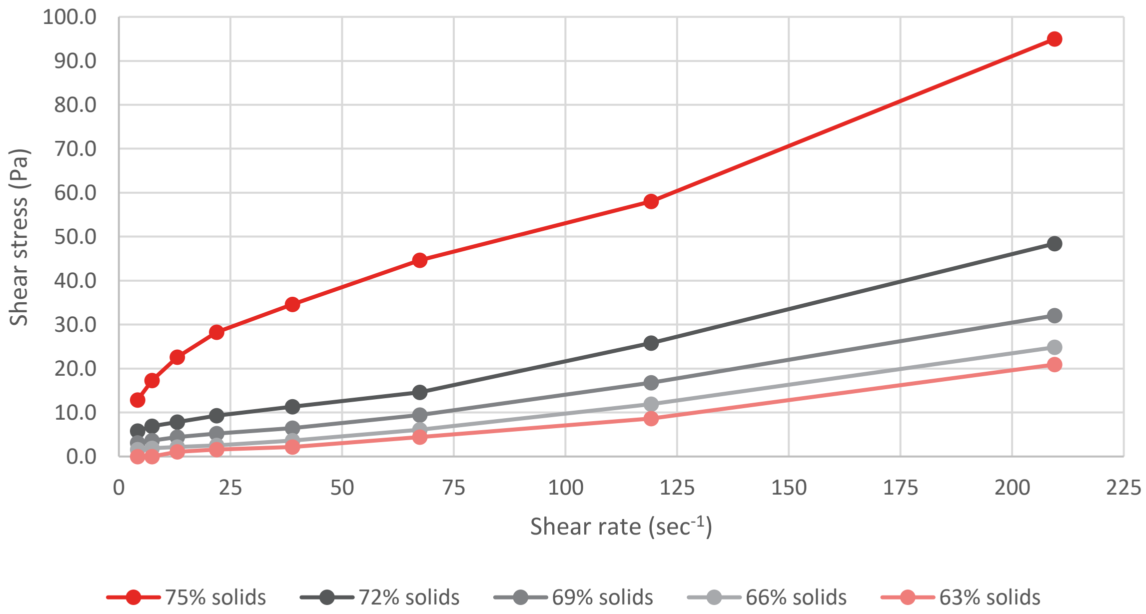


Figure 6-7 Shear stress Vs shear rate

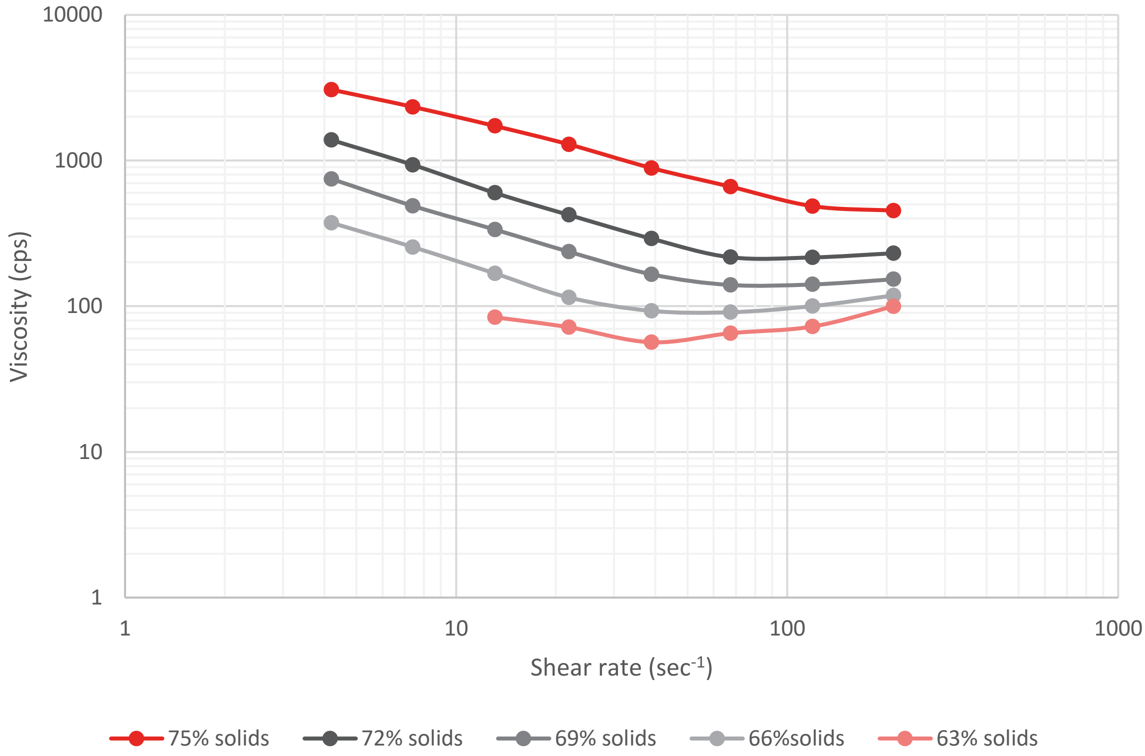


Figure 6-8 Viscosity Vs shear rate

6.2.13 Previous Work

Core and milled samples from the bulk Meknès composite (BMC), which is the composite used in previous testwork, were examined by optical microscopy. This analysis indicated that liberation commenced at 200 µm and that cassiterite whilst well liberated at 106 µm grind size to a limited extent remained partially or completely locked with tourmaline and quartz.

Mineralogical investigation of various middling and tail streams from gravity processing of the BMC sample indicated that middlings streams contained locked cassiterite and so required regrinding, and that tails streams contained fine locked cassiterite of grain size ranging from 5–25 µm.

6.2.14 QemScan Analysis

QemScan (quantitative evaluation of minerals by scanning) is an electron microscopy technique to automatically and rapidly analyse the mineralogy of metallurgical products, size-by-size and particle-by-particle. The detailed analysis establishes the percentage of each mineral in the sample, if each mineral is in an easily accessible form and how much mineral is recoverable economically.

A QemScan analysis of four variability samples in 2018 provided a quantitative analysis of cassiterite liberation by size (Figure 6-9 and Figure 6-10). This material was a combination of ore sorting accepts plus fines, and so the consequences of rejecting ~0.20% Sn material on the locking of cassiterite in the plant feed was of primary interest. The QemScan analysis showed that the liberation of cassiterite was very similar for both <150 µm and <106 µm material, indicating that grinding to less than 150 µm may not be necessary.

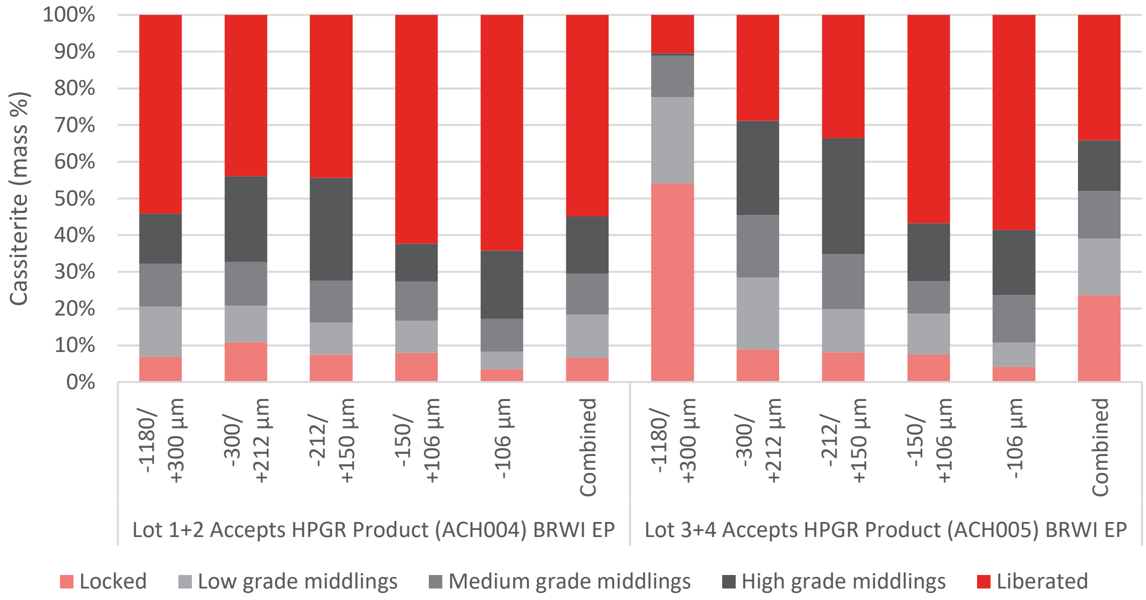


Figure 6-9 Cassiterite liberation lots 1 to 4

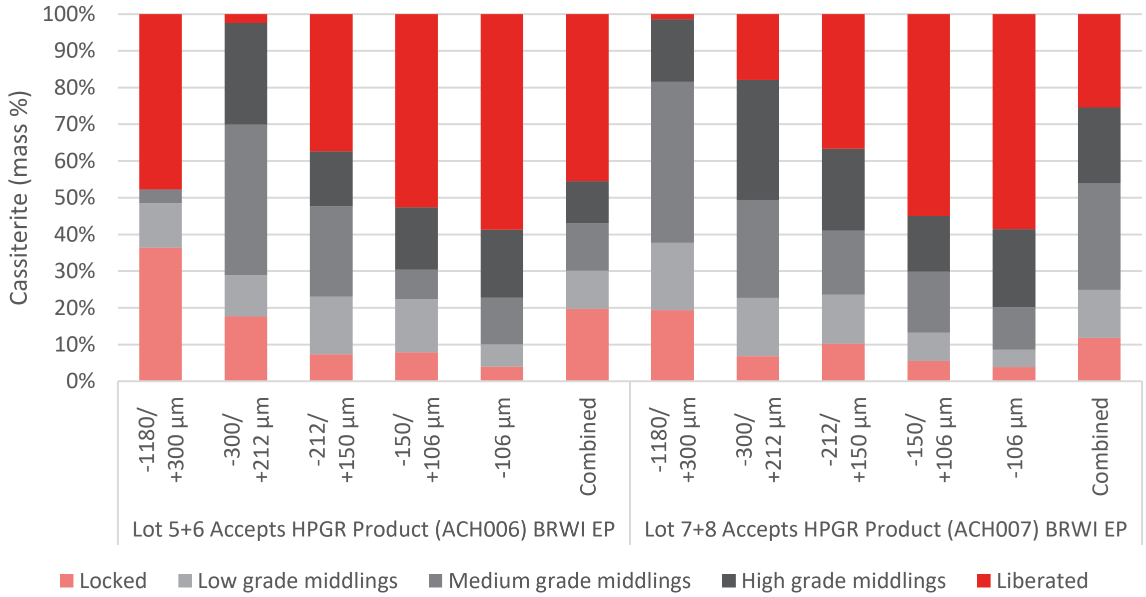


Figure 6-10 Cassiterite liberation lots 5 to 8

The analysis of the finer size range (Figure 6-11 and Figure 6-12) showed it consistently comprised 70% to 80% of liberated cassiterite or high grade middlings. It also showed that the fully locked (ie unrecoverable) cassiterite was less than 5%.

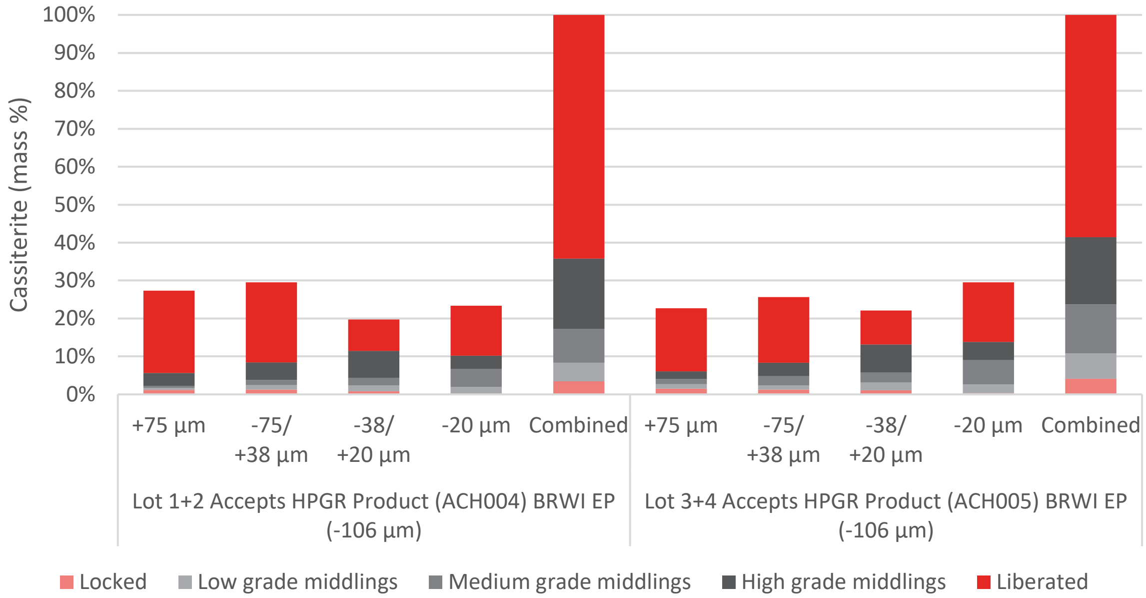


Figure 6-11 Cassiterite liberation by particle size for <106 µm fraction, lots 1-4

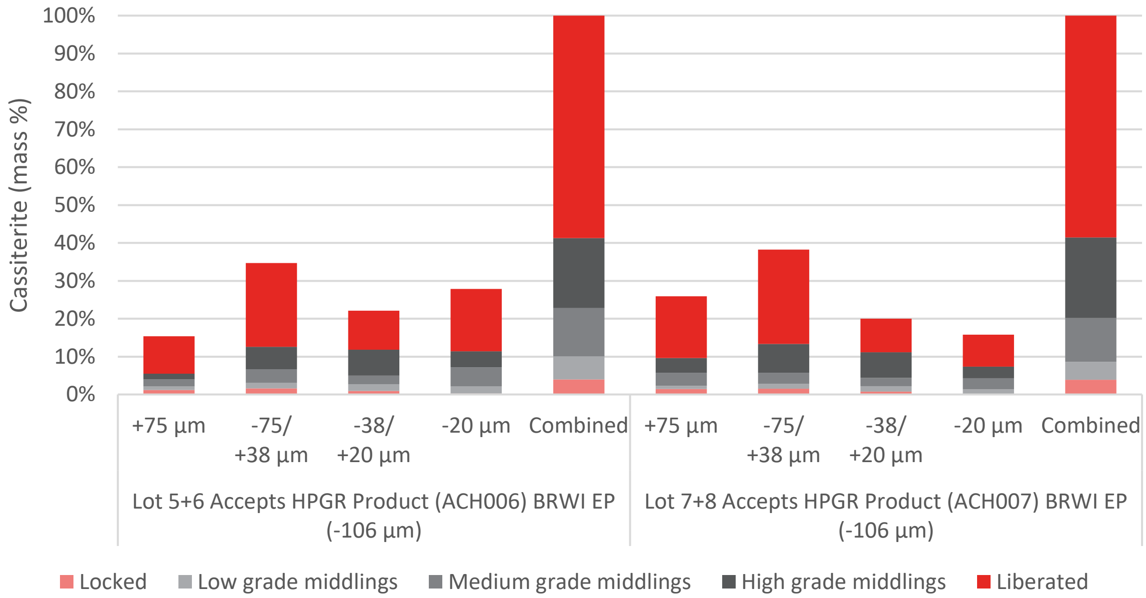


Figure 6-12 Cassiterite liberation by particle size for <106 µm fraction, lots 5-8

The outcome of this work is that:

- The four variability samples showed quite consistent cassiterite liberation characteristics.
- Between 55% and 60% of the cassiterite is fully liberated, with another 15% to 20% present as high-grade composites in the size range finer than 150 µm. This is consistent with the high gravity recovery achieved in test work and provides a basis for a high grade final product when gravity concentrates are dressed and combined with lower grade flotation concentrates.
- The fine gravity circuit incorporating a regrind stage is justified by the consistent ~30-40% of cassiterite in composites in the 38 µm to 150 µm size range where gravity separation is efficient.

- The population of low grade composites and fully locked cassiterite finer than 38 µm, which is the size range for flotation, is negligible at ~5% to 10% of the total.

These results were confirmed on a composite sample of the four ore zones by MODA (McArthur Ore Deposit Assessments Pty Ltd, located in Burnie, Tasmania) analysis using optical microscopy. Table 6-9 below shows that 69% of cassiterite is fully liberated, in particular in the 8 µm to 28 µm size range where 91% is liberated.

Table 6-9 Ore composite cassiterite distribution %

Fraction	Free	Binary with							Ternary+
		PyMa	Po	St	As	Cp	Feox	Ga	
>106µm	23	0	0	0	0	0	0	61	13
>53µm	49	0	0	0	0	0	0	50	1
>28µm	78	0	0	0	0	0	0	20	1
>8µm	91	0	0	0	0	0	0	9	0
TOTAL	69	0	0	0	0	0	0	28	2

PyMa=pyrite+marcasite, Po=pyrrhotite, St=stannite, As=arsenopyrite, Cp=chalcopyrite, Feox=goethite+magnetite+hematite, Ga=gangue. Tr means <0.5%. Rounding errors may occur.

In cassiterite only terms, the potential separation based on the observed liberation characteristics for a perfect process is shown in Figure 6-13 below.

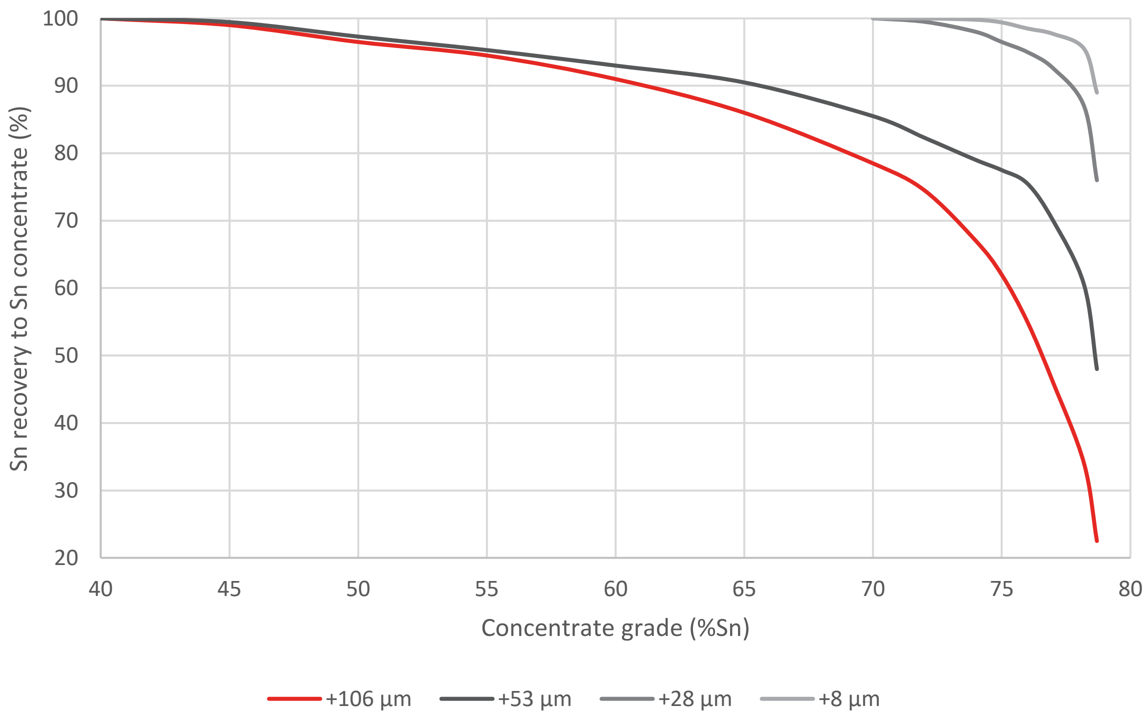


Figure 6-13 Ore composite – cumulative yield curves

6.2.15 Future Testwork

The single largest process loss is the slime tail; the material finer than 8 µm that does not respond well to flotation. Preliminary tests have been conducted using ultra fine (UF) Falcon concentrators to reject the light gangue slime particles followed by addition of the concentrate product to the cassiterite flotation stage. This successfully recovered half of the tin in slime tail without a major negative impact on flotation grade recovery performance, however further test work is required to enable a reliable design.

Further ore sorting testwork is planned to enable selection of the optimum technology for the Achmmach ore. The two major suppliers take a slightly different approach to the XRT separation, plus there are some technical innovations in development that could also have a positive impact. The technology is considered to be well established and reliable but will continue to evolve.

A new spiral technology has been developed in China which is reliably reported to provide superior performance at lower capital cost. This will be explored further at pilot scale as it may reduce both capital cost and the amount of tabling required.

Some aspects of the flotation testing, locked cycle gravity testing and the dressing of flotation concentrates by wet high intensity magnetic separation (WHIMS) will be incorporated into the final design. The potential impact of this work is only positive, with no risk of downside on the performance data utilised in this study.

6.2.16 Consideration of an Alternative Mass Balance

The low level of variability in ore types established by mineralogy and the results of various tests supports the validity of the earlier TTC testwork, where gravity and flotation testing was conducted in locked cycle on a shaft bulk sample taken from a stockpile remaining from an 80 m deep shaft and lateral development that was mined by BRPM circa 1990. The results of this testwork were striking in that gravity spiral tails in both coarse and fine circuits were very low grade, and so a large mass of this tail could be discarded. The cassiterite liberation profile shown in the mineralogy also supports the potential for discarding much larger quantities of spiral tail to final tail.

This has implications for the size of the regrind circuit and the flotation circuit. This in turn has implications for the slime tail loss and the cost incurred in fine gravity regrind and flotation. Both will reduce significantly given discard of the gravity tail streams and only regrinding of the middlings.

The flow sheet shown in this study has been developed conservatively, where only a small coarse gravity tail has been discarded. All the fine gravity tails are reground until passing 38µm and then advanced to flotation.

A mass balance has been developed which supports this scenario where flotation feed is reduced to 30% of the feed tonnage shown in this study. This potential improvement will be quantified in an addendum to the feasibility study, as the locked cycle gravity testing was only commissioned late in the test program when the low ore variability had been established.

6.3 Metallurgical Recovery Modelling

Metal recoveries obtained from testwork programs suggest a relationship between head grade and recovery, which is not an unusual phenomenon for the Achmmach style of mineralogy. It is likely that there is a limiting tails grade for the mineralisation that would represent intimate dispersion of cassiterite within the host rock caused by the hydrothermal nature of the original cassiterite mineralising event(s).

Examination of tails grades achievable through heavy liquid separation testing by CPG (June 2011) suggests that a background grade of about 0.15% Sn may exist for the Achmmach deposit. Given the nature of the occurrence of tin, it is plausible to expect cassiterite grain size to become coarser with increasing concentration, reflecting crystal deposition from higher strength geothermal fluid.

Grades below about 0.25% Sn appear to have limited availability of fine, free or partially locked cassiterite grains and recovery at this grade may be in the range 40% to 48% at best. Above the 0.25% Sn grade threshold, the incremental recovery may tend towards 85% as the cassiterite becomes increasingly responsive to conventional gravity separation techniques.

In 2017, testwork using the TTC process produced a concentrate grade of 55.3% Sn and average tin recovery over the life of mine of 74.6%; assuming an average feed grade of 0.82% Sn. This process was used in the design mass balance for the study. As shown in Figure 6-14 below, recovery at targeted concentrate grade will be proportional to feed grade.

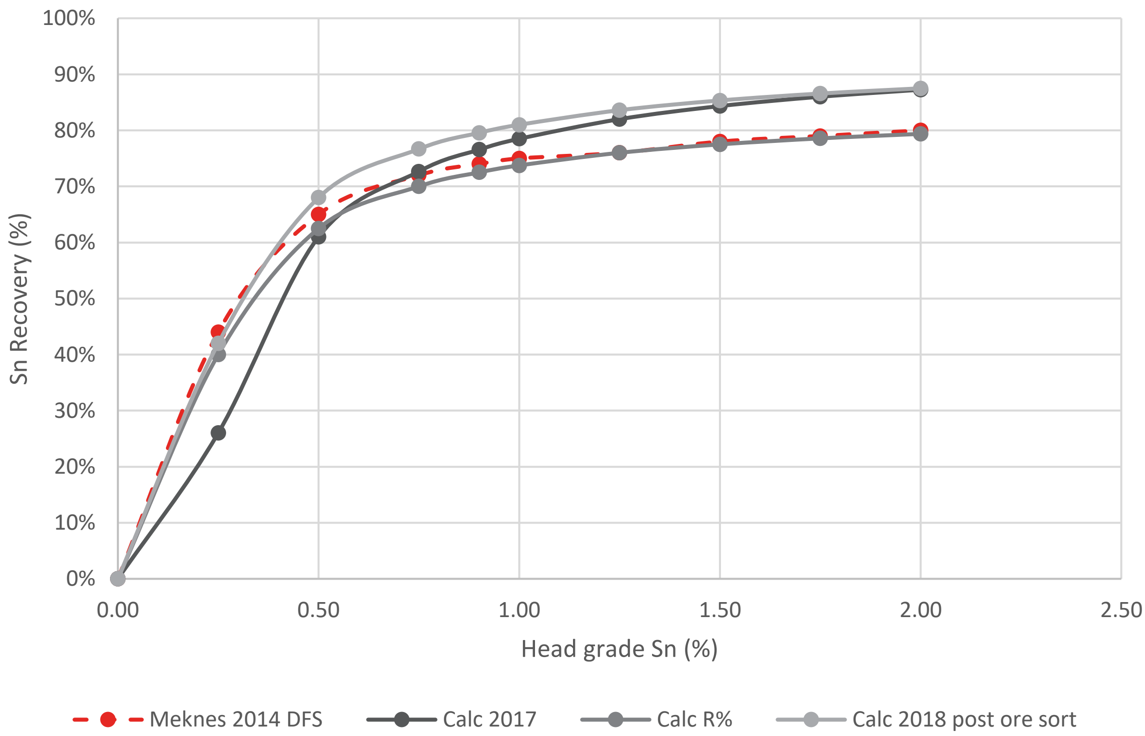


Figure 6-14 Head grade recovery models

The 'Calc 2018' relationship above is based on an escalating tail with feed grade and calibrated to the locked cycle gravity and flotation test results from the 2018 testwork and the 2016 TTC test programme.

Table 6-10 Head grade-recovery relationship

Head grade %Sn	Plant Recovery%	Tail grade %Sn
0.00	0	0.00
0.25	42	0.15
0.50	68	0.16
0.75	77	0.18
0.90	80	0.18
1.00	81	0.19
1.25	84	0.21
1.50	85	0.22
1.75	87	0.24
2.00	88	0.25

6.4 Metallurgical Design Concepts

6.4.1 Design Summary

The design of the treatment plant reflects:

- a simple and robust process flowsheet
- a flowsheet and plant layout that facilitates expansion
- a control philosophy that encompasses a reasonable level of instrumentation and control to help maintain steady operation and optimum throughput.

The key components of the flow sheet include:

- two stage crushing fed from the run of mine (ROM) stockpile by front end loader, crushing operation operated by the owner
- XRT ore sorting to reject 24% of the ROM material
- high pressure grinding rolls in open circuit (with recirculating ball mill scats) followed by closed circuit ball milling to a minimum P_{80} of 106 µm, and up to 150 µm
- gravity beneficiation for the coarse (plus 38 µm) fraction using spirals and tables followed by dressing using magnetic separation, sulphide flotation and tabling
- regrind and beneficiation in a fine gravity circuit for the 8 µm to 38 µm size fraction
- desliming, magnetic separation, sulphide flotation, attritioning, cassiterite flotation and dressing of concentrate with centrifugal gravity separation
- blending, filtration and bulk dispatch of final concentrate containing approximately 60% Sn
- tailings paste thickening
- reagent and consumables storage and reticulation and plant services.

6.4.2 Summary

The metallurgical response of the Achmmach ore to conventional gravity and flotation processes has been well established by several years of test work by Atlas Tin on composite samples.

The objective of the test work for the feasibility study was to reduce the capital and operating costs of the plant, to provide data in critical design areas on the influence of ore variability, and to seek means to enhance grade recovery performance.

The analysis of pre-concentration by ore sorting has proved a pivotal test program for the project. This is discussed fully in Appendix 6A along with the procedure for creating the representative samples required. The early elimination of gangue and the low-grade component from the ROM feed will fundamentally change not just processing but the broader aspects of the project by:

- upgrading the head grade of tin to the downstream processing plant
- reducing throughput capacity (and projected capital expenditure) of the downstream processing plant
- reducing total consumables, power and water requirements in downstream processing
- increasing the recovery of tin in the gravity and flotation circuit due to the higher feed grade achievable with ore sorting
- improving concentrate grades and so reducing concentrate shipping and treatment charges
- reducing tailings tonnage, tailings cost and environmental footprint
- providing ore sorter rejects for mine backfill or road base applications.

Toyota Tsusho Corporation recently developed proprietary technology for enhanced recovery of fine, liberated cassiterite in flotation. The process involves high energy attritioning after milling and use of a conventional reagent suite and some specialised depressants in flotation. This is described in more detail in the process plant section of this report. Laboratory scale test work was carried out under the supervision of an independent metallurgical consultant and demonstrated that significant improvement in both recovery and tin grade in concentrate could be achieved using the process as an addition to the flowsheet. LADP has relied on these results to prepare a LIMN mass balance simulation of the process which therefore also shows increased tin recovery.

The process plant design is based on two stage crushing and ore sorting followed by the use of high pressure grinding rolls and ball milling prior to gravity separation processes. The proposed flotation concentration technology is standard industry practice in terms of flow sheet and reagents but introduces a feed preparation step that is novel for tin projects but has been used in other base metal applications with success.

Given the relatively fine-grained nature of cassiterite in Achmmach ore, the use of cassiterite flotation and centrifugal gravity concentration will be more intensive than it may be in other hard rock tin recovery circuits. However, unlike some deposits, the occurrence of contaminant sulphide minerals is low, thus confining sulphide removal to simple steps ahead of gravity concentrate dressing and fine cassiterite flotation

6.5 Design Basis

6.5.1 Flowsheet Development

The original test work data generated for previous studies combined with the results from the 2018 program has enabled the development of a more cost effective and lower risk flowsheet.

Ore from underground is truck dumped on a run of mine (ROM) pad to both provide a buffer for steady production into the plant and to enable blending of ores to maintain a feed grade target. The 2018 test work indicates that the style of mineralisation is consistent along the strike of the orebody, so blending will be able to focus on grade rather than any major deviations in ore type.

The ore will be primary and secondary crushed using conventional crushers to a P_{80} of 32 mm in closed circuit to an open stockpile at 130 t/hr, requiring at this rate only 6,000 hours of operation per year, an average utilisation of 68%. This provides generous catch up capacity for periods of scheduled downtime.

The fine ore stockpile feeds the fines screen at 95 t/hr, removing all material <8 mm which constitutes 25% to 30% of the screen feed mass. The screen oversize feeds the ore sorter which rejects 40%+/-5% of the mass to a rejects stockpile, the mass split depending on the grade of the ore. This grade will be smoothed out by ROM pad blending. The rejects stockpile, at a grade generally less than 0.2% Sn, will be used underground for the cemented rock fill and for road works around site.

Steinert quote >95% availability for its ore sorting units in commercial operation. The nozzles in the pneumatic ejector systems do not cause significant downtime with development of start/shutdown automated purging systems, evolved nozzle design, upgraded compressor air intake filter systems and automatic firing/ejection tests.

The high grade accepts fraction from ore sorting combined with the <8 mm fines will be crushed in the HPGR at 68 t/hr. The HPGR discharge directly feeds the ball mill, and the mill scats from the mill trommel oversize, which have been estimated by mill circuit modelling to be small in tonnage, are recycled to the HPGR. The ball mill discharge is fed to the ball mill classifiers. The selected classifier strategy uses cyclones and screens in series in the ball mill circuit and a short axis grate discharge mill geometry, both contributing to the minimisation of the over-breakage of the very brittle cassiterite.

The HPGR is now considered mainstream crushing technology and typically has very high availability and power efficiency. For the Achmmach project, the HPGR has been thoroughly tested and has achieved all objectives, producing a finer size distribution than by conventional three stage crushing. Power consumption is low at the design pressure of 4,000 kN/m^2, and roll life is estimated to be 7,000 hours. This single unit has replaced the requirement for the third stage of crushing and rod milling. In effect the HPGR will have eliminated the rod mill cost and risk.

The study recovery flow sheet incorporates a two-stage conventional spiral/table gravity circuit, the primary to recovering liberated coarse cassiterite at >75 µm, the fine secondary circuit recovering cassiterite at 38 µm to 75 µm and includes a regrind circuit. Cassiterite finer than 38 µm is de-slimed, and the 8 µm to 38 µm cassiterite recovered by flotation. A proprietary feed preparation system enables flotation with high recovery of concentrates at 25% to 30% Sn, with cleaner tail streams recycled back through the feed preparation stage.

The low level of variability in ore types established by mineralogy and the results of various tests.

6.5.2 Crushing

Crushing of ore in two stages at 130 t/hr and 68% utilisation in closed circuit generates a 32mm P_{80} product which is well suited to the small roll diameter of the downstream HPGR, and to the reduction capability of the HPGR to generate an acceptable primary ball mill feed size. The HPGR diameter is largely a function of the machine size required for the throughput requirement. If the HPGR capacity is larger than required then part of the HPGR discharge can be recycled to HPGR feed to maintain choked feed conditions. This option has been conceptually included in the design and can be readily implemented if required.

The crushed product is fed to a 24 hour live load on ground stockpile, with ore reclaimed using twin vibrating feeders to a conveyor in a tunnel under the stockpile.

6.5.3 Ore Sorting

Fines screening is fed from the FOS at 95 t/hr and 90% utilisation removing ~25% to 30% of <8mm from the feed, and feeding >8mm screen oversize to ore sorting at 66 t/hr. The ore sorter typically rejects >40% of the >8mm feed on the basis of mineral content measured as a high density phase by the XRT system. The rejects are fired off the end of the sorting belt with compressed air. The ore sorter feed rate is controlled to a steady tonnage rate, with the difference in measurements of fines screen feed and undersize mass flow providing the process variable.

6.5.4 Grinding

The HPGR is fed at 68 t/hr with a combination of the ore sorter accepts, the <8mm fines, and a 15% recycle of ball mill scats (total 79 t/hr). The HPGR product is nominally 6mm P_{80} at the higher pressure settings.

The primary ball mill grinds the HPGR product to 80% passing 150 µm in closed circuit with cyclones and screens. The complex classifier system is selected to minimise the recycling of liberated cassiterite to mill feed. Should coarse material travel through the mill, around >6mm, then this will be removed from the mill discharge by the discharge trommel screen and recycled to the HPGR.

6.5.5 Coarse & Fine Gravity

The coarse gravity cyclone is fed at 68 t/hr and the >75 µm coarse gravity cyclone underflow is fed at 27 t/hr to the coarse gravity circuit where liberated cassiterite is recovered on spirals and tables. A final tail is taken from the spirals, and middling streams are directly fed to a regrind mill. The balance of <75 µm material is fed to the fine gravity classifier from where the <38 µm fraction is sent to desliming and the 38 µm to 75 µm fraction is fed to the fine spiral table circuit.

The fine gravity circuit is fed at 61 t/hr from the fine gravity cyclone underflow where liberated cassiterite is recovered on spirals and tables. Middlings from coarse and fine gravity are reground, and the regrind mill discharge is also fed back to the fine gravity cyclone. Slime formation in the regrind mill is minimised by feeding the reground material to spiral separation after only one pass through the mill.

The gravity concentrates are upgraded using magnetic separation to remove steel scrap, sulphide mineral is removed by flotation and tabling generates a final gravity concentrate. A small regrind mill is used to liberate composite cassiterite from table tails and the discharge recycled to table feed.

6.5.6 Cassiterite Flotation

The flotation process is fed at 38 to 42 t/hr and recovers cassiterite <38 µm. The <38 µm feed is prepared by desliming to remove <8 µm slimes, the deslime underflow by magnetic separation to remove steel scrap, then sulphide mineral removal by flotation, further desliming and then high intensity attritioning at low pH.

The cassiterite flotation rougher concentrate is fed to two stages of cleaning, with first cleaner tail recycled to the second stage of desliming. Scavenger concentrate can also be recycled with first cleaner tail.

The baseline option is to further upgrade the flotation concentrate by three stage Falcon centrifuges. The further upgrading of second cleaner flotation concentrate by wet high intensity magnetic separation is being investigated, which achieves this upgrade by removing weakly magnetic minerals that carry iron.

The <8 mm slimes constitute a significant loss of value, and with further testing a Falcon separation and flotation process will be further developed to reclaim ultrafine cassiterite from this stream.

6.5.7 Flotation Equipment Selection

In both sulphide flotation and cassiterite flotation the entrainment of fines is a problem for tin loss to sulphide concentrate and gangue contamination of cassiterite flotation concentrates respectively.

At the Minsur operation at San Rafael in Peru, and much earlier at Wheal Jane in UK, it has been clearly established that pneumatic cells with counter current froth washing gave significant benefits in terms of grade. This was especially important at San Rafael where the low selectivity sulpho-succinamate collector in use caused excessive contamination of cassiterite concentrates by gangue entrainment.

Consequently, the Imhoflot pneumatic cell has been specified for the Achmmach project. This operates on a similar principle to the Jameson cell with a pump driven aeration venturi and provides a high intensity collection zone. The configuration of the cell also enables the use of a spray bar on the froth phase as it moves to the discharge point. This is a much simpler arrangement than that required on a column cell which utilises a drip tray above the froth phase.

The cell cannot be tested at bench scale, but requires a pilot scale test bed. The cell has sufficient track record in terms of its basic functionality to warrant its adoption at this time.

6.6 Detailed Process Description

6.6.1 Introduction

The process description should be considered in conjunction with the following documents:

- design criteria and mass balance (Appendix 6B and Appendix 6C)
- equipment list (Appendix 6D)
- process flow diagrams and layout drawings (Appendix 6E and Appendix 6F).

The processing facility for the Achmmach project will produce tin concentrate assaying 60% Sn. The modern processing facility will separate the mineral cassiterite (SnO_2) from the gangue minerals using the following unit operations:

- crushing, and screening
- ore sorting
- primary grinding through High Pressure Grinding Rolls followed by a primary ball mill
- gravity separation and regrind
- de-slime and sulphide flotation
- cassiterite flotation
- flotation concentrate dressing
- gravity concentrate dressing (magnetic separation, sulphide flotation and tabling)
- combined concentrate filtration
- tailings thickening and disposal
- flotation & thickening reagents
- water and air services.

The design of the plant has been based on continuous operating regime.

The layout and elevations in the facility have been optimised to minimise pumping which would increase fines production and recovery losses. Where possible, slurry transfer by gravity has been adopted. The design of the processing flowsheets has also been influenced by the topography of the plant area and previously accepted building restrictions

6.6.2 Flow Sheet

The Block Flow Diagram in Figure 17.2 depicts the flow sheet developed for the feasibility study.

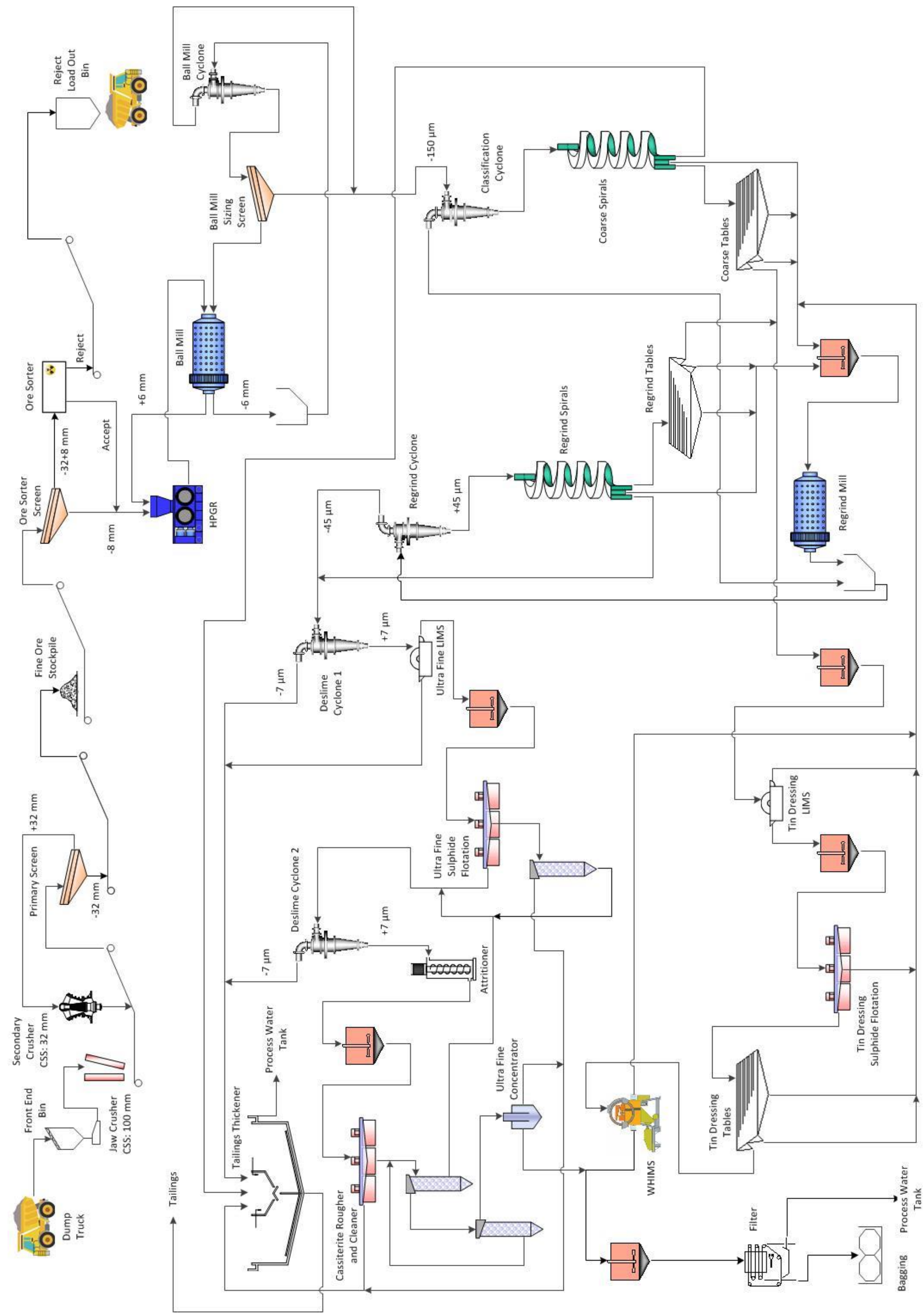


Figure 6-15 Process flow diagram

6.6.3 Plant Layout and General Process Description

Figure 6-16 is an extract from the preliminary general arrangement drawing of the process plant.

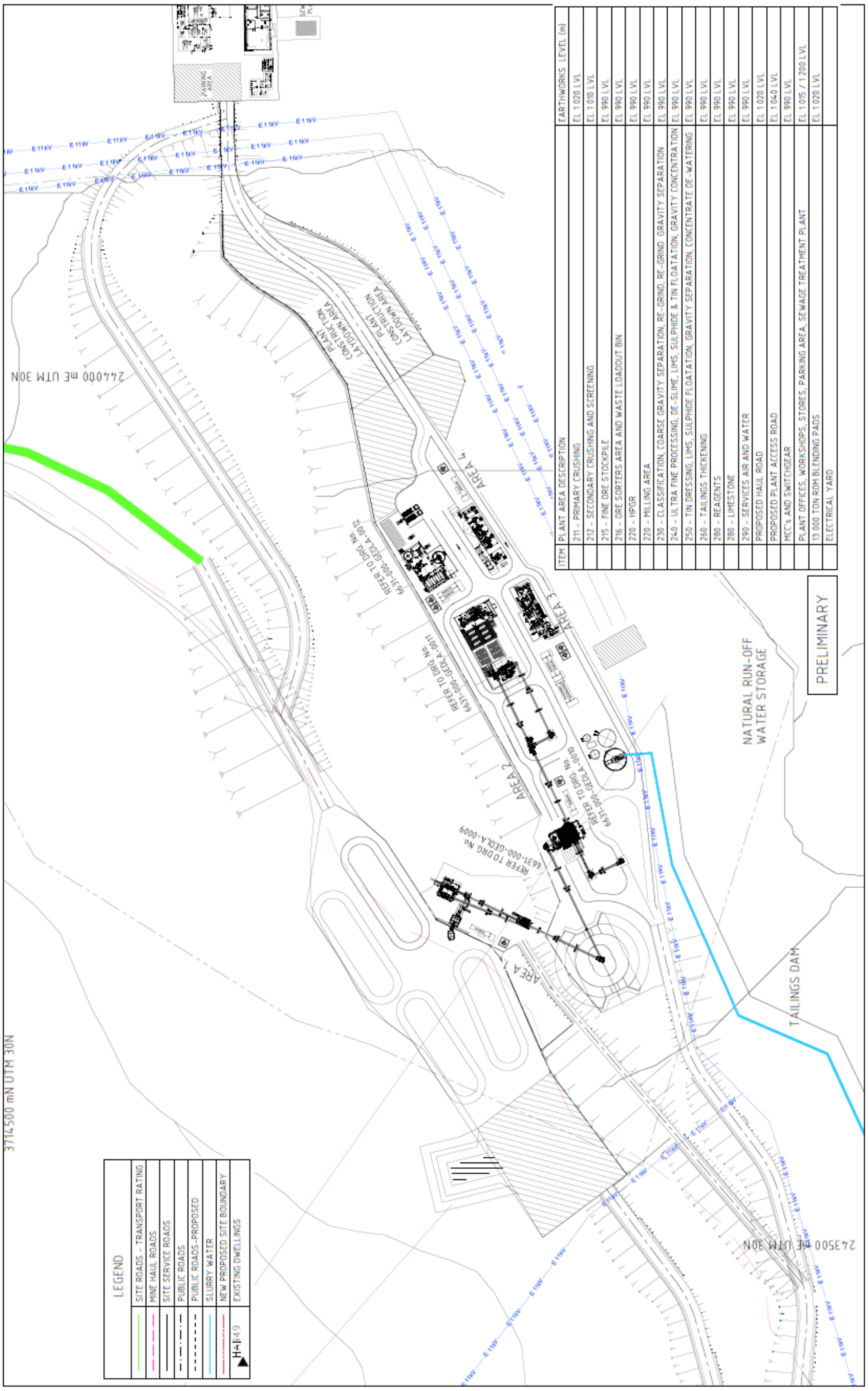


Figure 6-16 General arrangement of process plant

Run of mine (ROM) ore will be delivered by haul trucks and dumped on the ROM pad. Provision has been made for a ROM stockpile to allow feed blending to optimise plant performance. Management of the feed to the primary crusher to ensure a constant ore blend in the mill feed will be critical to achieving optimum throughput, recovery and tin production.

The processing plant is designed to process 750 ktpa of run of mine feed with an average grade of 0.9% tin (Sn). With the addition of two ore sorting units, approximately 30% of the overall feed tonnage is rejected resulting in approximately 530 ktpa at a higher grade reporting to grinding and downstream processes. The output of the plant is approximately 4,500 tpa of tin in concentrate.

The ore will be crushed from a top size of 600 mm to a P_{100} of nominally 32 mm in a two-stage crushing and screening plant and will be conveyed to a live fine ore stockpile with 24 hours capacity. The crushed ore will then be withdrawn from the stockpile via a feeder and passed over an 8 mm screen with the 8 mm to 32 mm fraction reporting to two ore sorting units. The ore sorters will reject about 40% of their feed (30% of the crushed ore) onto a stockpile with the accepts reporting to an open circuit HPGR unit. The HPGR product will then be milled to a P_{80} of nominally 106 µm using a ball mill in closed circuit operation with cyclones and stacked classification screens.

The mill product will be further cyclone classified at 75 µm with the underflow reporting to a coarse gravity circuit and the overflow reporting to the regrind mill discharge sump prior to further classification at 38 µm. The >38µm milled ore will be beneficiated using spirals and shaking tables to produce a 20% to 25% tin gravity concentrate. The spiral and table middlings will be reground in the regrind ball mill. The gravity concentrate will be further upgraded to >55% tin using magnetic separation, sulphide flotation and final cleaning shaking tables in the tin dressing circuit.

The >38µm fraction created from the primary ball mill and regrind mill will be de-slimed at nominally 8µm. The 8 µm to 38 µm fraction will be prepared for cassiterite flotation using low intensity magnetic separation, sulphide flotation and attritioning. The concentrate from cassiterite flotation will be dressed using centrifugal gravity concentrators to achieve a 50% tin, low grade concentrate. The <8µm fraction cannot be processed and will report to tailings.

The gravity tin dressing concentrate and ultrafine flotation concentrate will be combined, filtered and bagged for transportation to an offshore smelter.

The selection of process equipment was based on test work, basic metallurgical principles, vendor information and bench marking against similar existing tin beneficiation plants. The key equipment design decisions are explained in the following sections.

Table 6-11 below shows the basic process parameters for the overall plant flowsheet.

Table 6-11 Plant process parameters

Parameter	Value	Unit
Overall		
Plant Throughput (initial)	750,000	tpa
Head Grade (average)	0.82	% Sn
Operating Hours		
Crushing Plant	5,957	hrs/annum
Milling and beneficiation Plant	7,906	hrs/annum
Coarse Concentrate		
Grade	>55	% Sn
Rate	0.73	t/hr
Ultra Fine Concentrate		
Grade	55.74	% Sn
Rate	0.44	t/hr
Overall Concentrate		
Grade	60	% Sn
Rate	1.2	t/hr
Recovery	77.2	%

6.6.4 Comminution and Sorting

Crushing

The crushing circuit will include a primary jaw crusher operating in open circuit. Jaw crusher product will be conveyed to a secondary cone crusher stage operating in closed circuit with a screen cutting at a nominal 32 mm. Screen undersize will report to the fine ore stockpile. Overall, the objective of the crushing circuit is to be to reduce the top size of the ore from about 600 mm to P_{80} of 32 mm. Due to the highly abrasive nature of the ore, quick change wear components have been specified.

Figure 6-17 shows the process flow diagram for the crushing circuit as developed during the study.

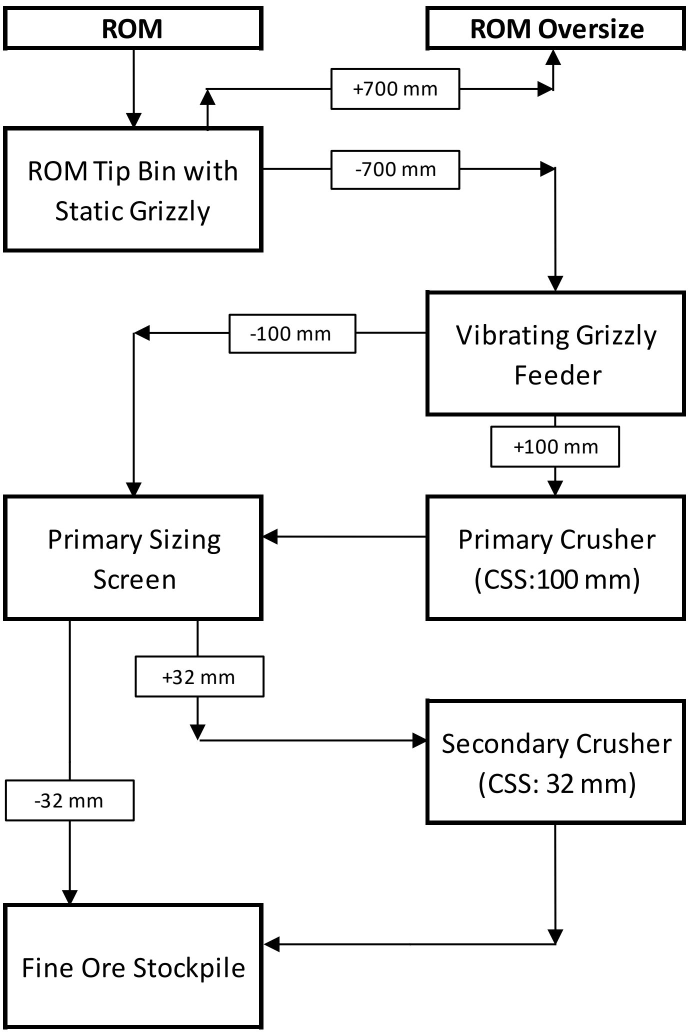


Figure 6-17 Crushing circuit flow diagram

The crusher plant design criteria displayed in Table 6-12 highlight the characteristics of the Achmmach ore, which is very abrasive and will require a large amount of energy to perform a size reduction.

The crushing circuit will operate for 18 hours a day to take advantage of the lower power cost during non-peak hours.

Table 6-12 Crushing circuit design parameters

Design parameter	Value	Unit
Operating hours per year	5,957	hrs/annum
Crushing plant feed	150	tph
Solids density	2.86-2.90	t/m^3
Bulk density	1.8	t/m^3
Crusher work index	32	kWh/t
Unconfined compressive strength	228	MPa
Abrasion index	0.73	-

SMC Test		
Mic	12.7	kWh/t
A	73.5	-
b	0.29	-
A x b	24.3	-

The circuit comprises a primary crushing building with a rom bin preceded by a static grizzly screen. A vibrating grizzly feeder will feed ore to a primary jaw crusher. The primary crushed ore will be conveyed over a weightometer to the primary sizing screen screening at 32 mm. Screen oversize will be conveyed to the secondary crusher feed bin and be fed to the secondary crusher via a feeder. A belt magnet) installed on the secondary crusher feed conveyor will protect the circuit from tramp iron. Crusher product will report to the primary sizing screen feed conveyor for re-sizing. The sizing screen undersize (<32 mm) will be conveyed to the fine ore stockpile.

The secondary crusher is equipped with a feed bin and vibrating feeder to control surging within the circuit.

Sorting

The crushed ore is control fed to the ore sorter sizing screen feed conveyor feed conveyor over a weightometer and feeds the ore sorter sizing screen which is designed to cut at 8 mm. The screen oversize gravity feeds the ore sorter feed bin which has a buffer capacity of 30 minutes. Two pan feeders withdraw the sized material from the bin and feed the two ore sorting units at a controlled rate. The ore sorters eject lumps of ore containing little or no tin and these report to the reject stockpile conveyor which feeds the reject stockpile. A weightometer measures the amount of reject material for accounting purposes.

HPGR

The grinding circuit design is critical to maximise the final cassiterite recovery. The ore is hard and abrasive with testwork indicating a bond ball mill index of 24 kW/t and an abrasion index in the range. Unlike the host rock, cassiterite is friable and care must be taken during grinding to minimise slime particle production. The HPGR testwork showed favourable results in maximising the deportment of >38 µm particles to the cassiterite flotation feed.

The accepted ore from the ore sorting units is fed onto the HPGR feed bin conveyor together with the sizing screen undersize (<8 mm material). Two weightometers measure the two streams feeding this conveyor and a belt magnet is positioned above the conveyor to remove any tramp metal that may cause damage to the HPGR downstream. Sized and sorted ore reports to the HPGR feed bin before being fed via feeder to the HPGR, the further crushed product from which falls onto the ball mill feed conveyor. A weightometer measures the fresh feed to the ball mill.

Milling

High frequency fine screens were selected to close the mill circuit in the 2017 study and after control and cost considerations a hydrocyclone cluster was inserted into the flowsheet ahead of the fine screens to take load off these units. Although less costly, cyclones as classification units allow high specific gravity materials to build up in the mill circulating load resulting in overgrinding of these higher

SG minerals. Fine screens achieve solely a size separation, independent of particle density, which maximises coarse liberated cassiterite recovery and minimises over-grinding. A 150µm screen aperture was selected to achieve the target grind size.

The grind product size of 80% passing 106 µm minimum to 150 µm was selected as it is the optimum size at which a significant fraction of cassiterite becomes liberated. Mineralogical test work indicates that the majority of cassiterite has a grain size of approximately 53 µm. A grind size that is finer than 125 µm would result in more fines generation and loss of liberated cassiterite during flotation. The 125µm target grind also gives the flexibility to grind coarser to 150 µm or finer to 106 µm as the ore varies over the life of the mine (screen panel changes will facilitate this flexibility).

Figure 6-18 illustrates the configuration of the milling circuit.

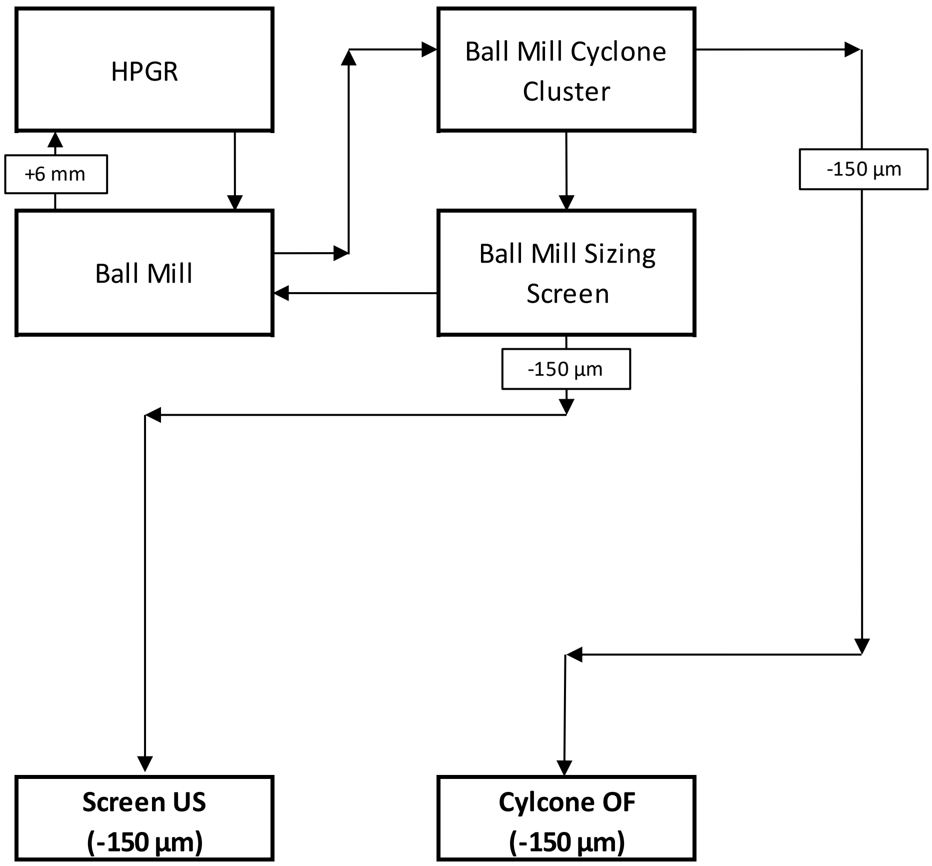


Figure 6-18 Primary milling circuit

Mill feed will enter the ball mill feed chute where it will be diluted to the correct pulp density for milling. The ball mill will be fitted with a trommel to remove scats and stray mill balls which will either report to a bunker or be conveyed back to the HPGR. New mill balls will be added to the ball mill feed chute via a kibble fed via a hoist.

Ball mill product slurry will be pumped from the mill product hopper to the ball mill cyclone cluster. The ball mill cyclone overflow (<106 µm) will gravitate to the mill product hopper and cyclone underflow reporting to the sizing screen distribution box before entering the fine screens. Screen undersize will join the cyclone cluster overflow in the mill product hopper and the screen oversize with screen oversize reporting back to the ball mill.

The slurry will be diluted in the mill discharge hopper and then pumped to the mill which will then gravitate to the ball mill sizing screens. Five screens with 150 µm apertures will be in closed circuit with the secondary ball mill. Screen oversize will report to the secondary ball mill and screen undersize will report to the mill product hopper.

Minus 106µm slurry will be pumped from the mill product hopper to the classification cyclone cluster in the spiral building. The classification cyclone overflow (<75 µm) will gravitate to the regrind mill discharge hopper for pumping to the regrind classification cyclone cluster. Primary classification cyclone underflow (150 x 75 µm) will gravitate to the coarse spiral circuit.

6.6.5 Coarse and Fine Gravity Separation

Gravity Spirals and Tables

The design includes separate coarse and regrind spiral circuits each utilising rougher, middlings and cleaner spirals followed by re-cleaner tables.

The coarse (75 µm to 150 µm) spiral circuit consists of rougher, middlings and cleaner banks of spirals with re-cleaner tables to produce a final concentrate suitable as feed to tin dressing. The minerals present in the spiral feed are cassiterite, major gangue minerals quartz and tourmaline as well as sulphides including pyrite, pyrrhotite and arsenopyrite. The aim of the coarse spiral circuit is to separate the heavy cassiterite and sulphides to concentrate, with the non-liberated cassiterite and gangue particles going to middlings and the free gangue to tailings.

The 75 to 125 µm coarse spiral feed has a size fraction range and quantity of near size material common to standard spiralling. This allows the use of spirals that are standard throughout industry and have been proven on Achmmach Tin Project ore by test work. The coarse cleaner spiral will be a reduced capacity cleaner type spiral to further increase efficiency. All coarse spirals will be twin start spirals to allow more access for optimisation and to increase maintainability. The coarse spiral feed distributor boxes will be fitted with an overflow to guarantee constant level to reduce the potential for performance variation.

Figure 6-19 is a block diagram of the coarse gravity separation circuit.

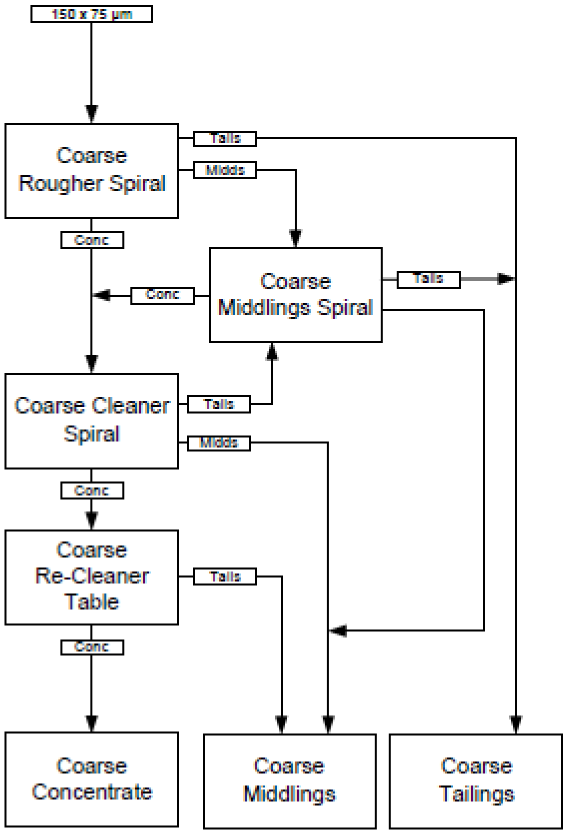


Figure 6-19 Coarse gravity separation

The coarse spiral feed slurry will be diluted in the coarse rougher feed hopper to the appropriate spiral feed density and pumped to the coarse rougher spiral bank. Coarse rougher concentrate reports to the coarse cleaner feed hopper, coarse rougher middlings reports to the coarse middlings feed hopper and coarse rougher tailings reports to the spiral tailings feed hopper.

The coarse middlings spiral feed containing misplaced gangue and valuable minerals will be diluted to spiral feed density and pumped to the coarse middlings spiral bank. Coarse middlings concentrate reports to the coarse cleaner feed hopper, coarse middlings reports to the spiral middlings feed hopper and coarse middlings tailings reports to the spiral tailings feed hopper.

The coarse cleaner feed slurry will contain mostly free cassiterite and sulphides, heavier middlings and some misplaced light particles. The slurry is diluted to the correct spiral feed density and pumped to the coarse cleaner spiral bank. Coarse cleaner concentrate reports to the coarse re-cleaner tables, coarse cleaner middlings report to the spiral middlings feed hopper and coarse cleaner tailings reports to the coarse middlings feed hopper.

The coarse cleaner table feed will contain mostly free cassiterite, sulphides and heavy middlings. Table concentrate drops or is flushed into the gravity concentrate hopper. The concentrate will be pumped to the tin dressing area. The coarse table tailings are pumped to the spiral middlings feed hopper and are then transferred to the spiral middlings regrind circuit.

Regrind milling and gravity separation

As with the primary circuit, the regrind mill design is critical to maximise liberation while minimising cassiterite slimes creation. The gravity middlings have an P_{80} of 120 µm. The design requires a grind product P_{80} of 38 µm. The milling product size was chosen to allow liberated cassiterite to be processed with spirals before being over-ground. This maximises cassiterite recovery in the regrind spiral plant and reduces the quantity of cassiterite reporting to ultrafine processing. This design decision was endorsed by bench scale test work.

In previous studies various types of fine grinding mills were considered for the Achmmach duty which requires predominantly attrition milling:

- IsaMill
- stirred vertimill
- conventional ball mill.

The Isa mill and the stirred mill are both more energy efficient for the required duty than a conventional ball mill but have significantly higher capital costs. A horizontal ball mill was therefore selected for this application. The mill will enable enhanced grind control.

The regrind mill flowsheet utilises a closed circuit in which mill discharge is classified via cyclones with cyclone underflow reporting to a separate spiral and table processing circuit. This arrangement allows the liberated cassiterite that preferentially reports to cyclone underflow to be recovered before over-grinding.

The regrind ball mill will be fed with spiral middlings from the coarse and regrind gravity circuits via the agitated regrind surge tank. The pulp will be pumped to the mill after dilution to the correct density. The regrind mill is fitted with a trommel to scalp off expected milling media to protect the downstream equipment. Trommel oversize will be directed to a bunker for removal. Reground ore will pass through the trommel to the regrind mill discharge hopper. The slurry will be then pumped to the regrind classification cyclone cluster.

If the regrind milling and gravity circuit is not operational for a period greater than an hour the spiral middlings can be directed to the tailings thickener in order to keep the rest of the plant in operation.

Cyclone underflow will report to the regrind gravity circuit and cyclone overflow will report to the ultrafines processing area.

New mill balls will be added to the regrind mill feed chute via a kibble fed via a hoist and magnet.

The 38 to 75 µm fine fraction is more difficult to separate than the coarse fraction. The spiral type chosen for the regrind processing is a low capacity specialised fine spiral. These spirals are not commonly used in industry but have been proven on Achmmach ore by test work. As with the coarse spirals, these spirals will be twin start to increase access and operability as well as have an overflow to increase stability.

The spiral launders used in the Achmmach plant will stretch along the length of the bank and discharge in a centralised piping corridor. This greatly reduces the complexity of the launder piping, reducing the difficulty of redirecting steams and gives access for sampling and flushing.

Figure 6-20 is a block diagram of the regrind gravity separation circuit.

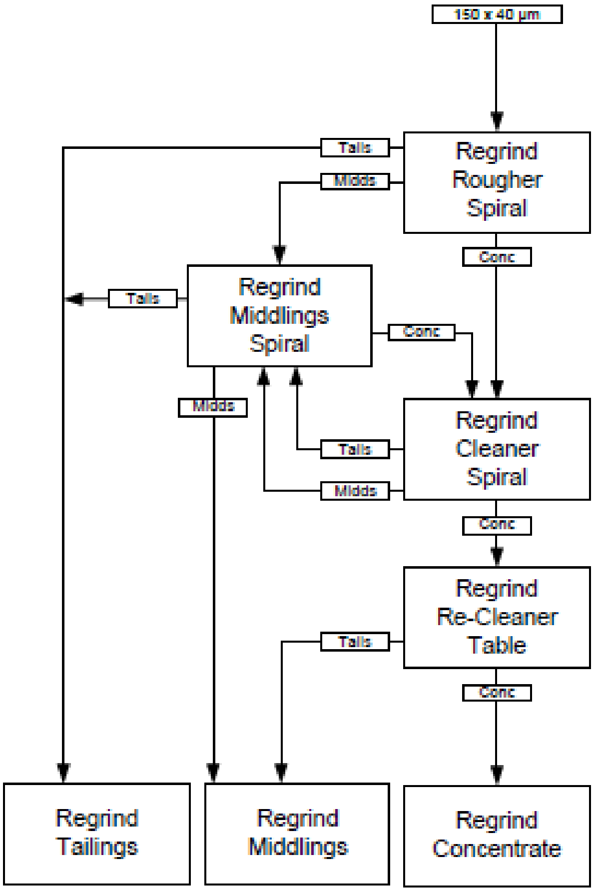


Figure 6-20 Regrind gravity separation

The purpose of the regrind classification cyclone cluster is to remove the ultrafine <38 µm material that is too fine too beneficiate in the regrind spiral circuit. The cyclone overflow will gravitate to the primary de-slime feed hopper and then will be pumped to the primary deslime cyclone cluster.

The regrind spiral circuit feed slurry will be diluted in the regrind rougher spiral hopper to the appropriate spiral feed density and gravity fed to the regrind rougher spiral bank. Regrind rougher concentrate will report to the regrind cleaner feed hopper, regrind rougher middlings will report to the regrind middlings feed hopper and regrind rougher tailings will report via cyclone protection screens to the primary deslime feed hopper.

The regrind middlings will contain mostly free gangue, non-liberated middlings ultrafine cassiterite and some misplaced fine cassiterite. The slurry will be diluted to spiral feed density and pumped to the regrind middlings spirals. The regrind middlings concentrate will report to the regrind cleaner feed hopper, middlings will be combined with regrind cleaner tails and recycled to the regrind surge tank ahead of regrind milling and regrind middlings tailings will report to the ultrafines processing circuit.

The regrind cleaner feed stream will contain mostly free cassiterite and sulphides, heavier middlings and some misplaced light particles. The slurry will be diluted to the correct spiral feed density and pumped to the regrind cleaner spirals. The regrind cleaner concentrate will report to the regrind re-cleaner tables and the regrind cleaner middlings will be recycled to the regrind middlings hopper.

The regrind cleaner table feed will contain mostly free cassiterite, sulphides and heavy middlings. Table concentrate drops into the gravity concentrate launder to be pumped via the gravity concentrate hopper to the tin dressing area 250. The regrind table tailings will be pumped to the regrind rougher surge tank.

Desliming

Hydraulic and surface tension forces have more of an effect on particles finer than 5 µm than gravity. This causes increased misplacement in flotation and other separation processes. In some cases depressants are available to reduce the slime carry over, but in the case of the Achmmach cassiterite flotation circuit the depressants available are not selective enough to achieve acceptable rejection of minus 8 µm material. This necessitates de-sliming before both sulphide and cassiterite flotation.

Slimes removed from the ultrafines contain cassiterite. It is therefore essential to reduce the mass in the slimes fraction and hence the quantity of cassiterite lost. A cut point of 8 µm (P_{80} cyclone overflow) was selected as this was shown in test work to facilitate acceptable flotation performance.

An 8 µm cyclone cut point is a difficult duty to achieve and requires a number of small cyclones. The design utilises a two stage circuit where the primary cyclone underflow is re-cycloned to improve de-sliming efficiency. Primary de-sliming, cutting at 8 µm removes the majority of the underflow mass and reduces the duty of the secondary cycloning stage.

The primary duty requires 37 to 60 mm cyclones operating at 335 kPa, with a similar cluster needed for the secondary desliming operation. Such small cyclones need protection from oversize material to minimise the chance of blockages and sieve bends will be used to achieve this.

All ultrafine material created in the primary and regrind milling will report to the ultrafine processing area. This material will report to primary de-sliming with cyclone underflow being directed to magnetic separation and ultrafine sulphide flotation. Tailings from this circuit will then be directed to secondary de-sliming, attritioning and conditioning, cassiterite flotation and ultrafine gravity concentration to achieve a saleable product.

The ultrafine processing area will be fitted with a bypass to the tailings thickener. This decouples the area from the primary mills and will allow the milling and gravity separation to operate while the ultrafine plant is down for maintenance.

Sulphide Removal

All material requiring ultrafine treatment will report to the primary deslime feed hopper in the spiral building. Cyclone protection screens are situated above the hopper to minimise the potential of foreign objects blocking either the primary or secondary de-sliming cyclones. The ultrafine feed will be pumped to the primary de-slime cyclone cluster. The primary de-slime cyclones will classify the ultrafines, targeting a ‘cut point’ of 8 µm. Cyclone overflow will gravitate to the ultrafine sulphide hopper before being pumped to the tailings thickener. Primary cyclone underflow will flow by gravity to the ultrafine LIMS low intensity magnetic separator. Magnetic concentrate containing pyrrhotite as well as any tramp iron will flow to the ultrafine sulphide hopper while the non-magnetic stream will be routed to the agitated ultrafine sulphide conditioning tank.

In ultrafine sulphide flotation the sulphide species such as pyrite and arsenopyrite will report to concentrate and the cassiterite and other oxides will report to tailings. Sodium isobutyl xanthate (collector) will be added to the conditioning tanks. Methyl isobutyl carbinol (frother) is dosed directly into the cells. The non-magnetic pulp will be pumped to the reverse sulphide flotation circuit. A single pneumatic ultrafine sulphide flotation cell will comprise the rougher circuit, to be followed by a single cleaner cell. Cleaner concentrate containing non-magnetic sulphides will be combined with the magnetic pyrrhotite concentrate and pumped to the TSF.

Secondary Desliming and Cassiterite Flotation

After sulphide flotation the ultrafines will enter the cassiterite flotation circuit. Early mineralogical and bench scale test work carried out on Achmmach material nearly ten years ago suggested that tin in ultrafines was not liberated even at 8 µm and that ultrafine milling would be needed to liberate the fine cassiterite grains. However, more recent QemScan work revealed that cassiterite in the 8 to 38 µm size fraction is almost all available for recovery by flotation.

The Achmmach cassiterite flotation is a standard flotation process flow sheet using Imhoflot pneumatic flotation cells to separate fine cassiterite from gangue minerals. The similarity of the surface chemistry of the different mineral species typically makes the separation of cassiterite difficult. In this case, the recommended cassiterite flotation circuit utilises rougher and scavenger flotation steps before two stages of cleaner flotation. This is a closed circuit with cleaner tailings being recycled. Testwork conducted by TTC demonstrated that the collector SPA in conjunction with SSF gangue depressant is superior to the TX5 collector originally utilized.

The ultra-fine sulphide combined rougher and cleaner tails will report to the secondary de-slime feed hopper and will then be pumped to the secondary de-slime cyclone cluster. The secondary cyclones will also target a ‘cut point’ of 8 µm with the aim of maximizing available feed in cyclone underflow to downstream tin recovery after removal of the majority of the slimes which will affect flotation performance. The secondary de-slime cyclone’s overflow cannot be processed and will report to the tailings thickener.

The secondary cyclone underflow from desliming will be gravity fed to the ultrafine attritioner feed hopper and pumped to the ultrafine high energy attritioner. The ultrafine material discharges into another feed hopper and is pumped to the agitated cassiterite rougher conditioning tanks. Here sulphuric acid (pH modifier) and SSF (activator) will be added to help clean mineral surfaces.

After conditioning, the slurry will be pumped to the cassiterite rougher flotation cells. The frother methylisobutyl carbinol (MIBC) will be added directly to these cells when required. Rougher tailings will report to the cassiterite flotation tailings hopper and the rougher concentrate will be pumped to the cassiterite cleaner one flotation cell. Tailings from cleaner cell one will report back to the attritioner feed sump and cleaner concentrate will report to the cassiterite cleaner two flotation cell. Tailings from cleaner cell two will be recycled to cleaner cell one and the recleaner concentrate will report to the ultra-fine concentrator feed tank.

Figure 6-21 is based on the SSO report of 2016 and illustrates a typical flotation circuit of the correct configuration using Imhoflot pneumatic flotation cells.

Table 6-13 summarises float reagents types, duty and dose rates, based on the latest available test work.

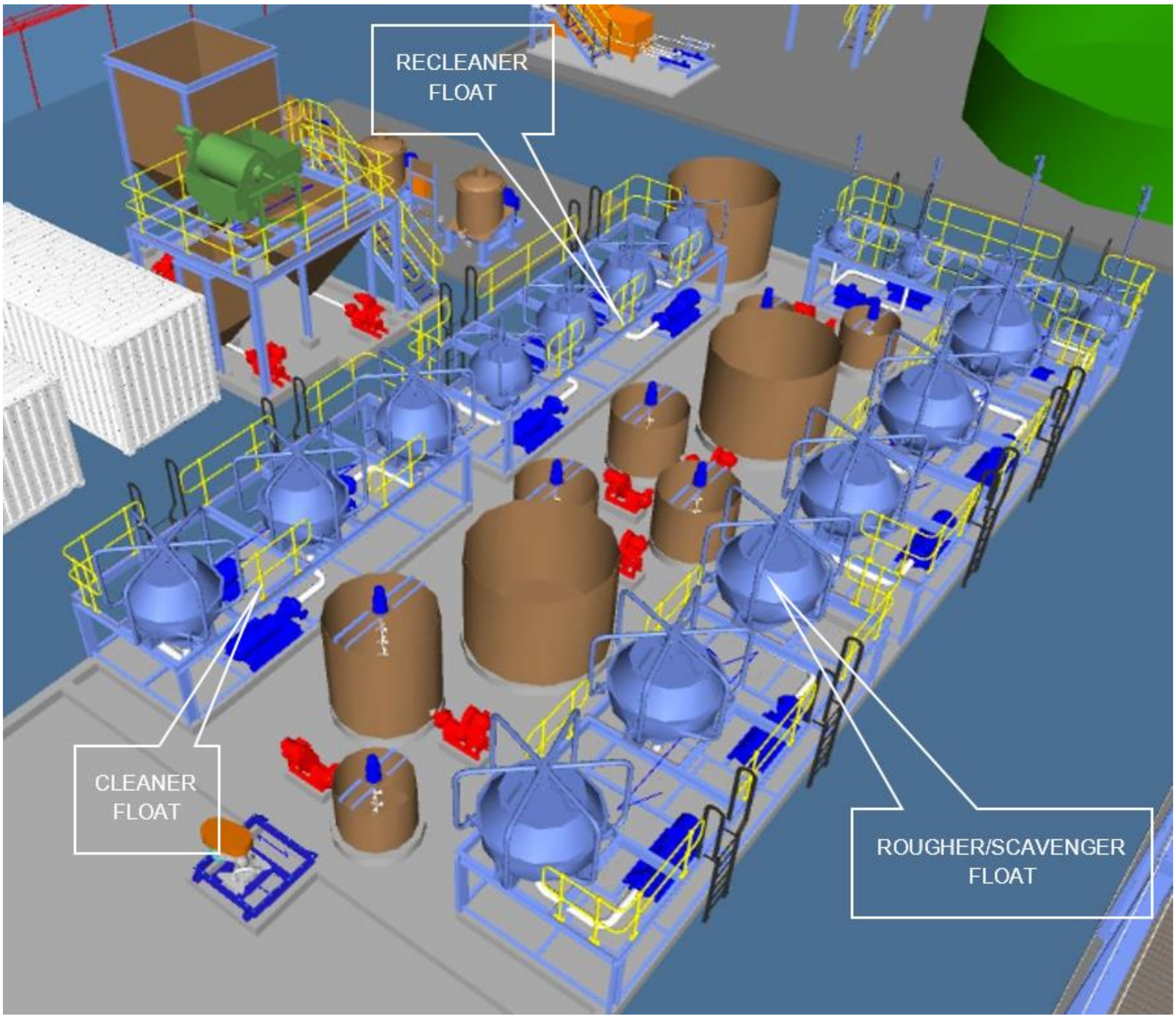


Figure 6-21 Flotation circuit

Table 6-13 Flotation reagents

Reagent	Function	Dose Rate, g/t ROM Ore
Methyl iso-butyl carbamate (MIBC)	Frother	18
Sodium isopropyl xanthate (SIPX)	Collector-sulphides	117
SSF Modifier	Surface conditioner	246
SPA/OPS30	Collector – cassiterite	250
H57	Frother 2	30
Fluo-silicic Acid	Depressant-silicates	246
Sulphuric Acid	pH Modifier	'246+ 246

Ultrafine Gravity Separation

Due to the selectivity of the cassiterite flotation reagents, both cassiterite and some gangue species will report to the cassiterite flotation circuit concentrate. Due to the density difference between cassiterite and the remaining silicates, ultrafine gravity concentrators are the logical processing step for the final upgrade to produce a saleable concentrate.

The possible equipment within the ultrafine concentrator category included:

- Knelson concentrators
- Falcon concentrators
- Kelsey jigs
- Mozley MGS.

Established tin operations now use ultrafine Falcon concentrators instead of the Mozley MGS and Kelsey Jig units. The Falcon has typically been found to be superior when comparing capital and operating costs and the ability to run continuously.

The design in this case consequently utilises a rougher, cleaner and scavenger ultrafine Falcon circuit to upgrade the cassiterite flotation concentrate. The concentrators will be gravity fed from a header box above each stage. This will allow the units to be isolated when in the flushing stage of operation.

Flotation cell cleaner two concentrate will gravitate to the agitated ultrafine rougher concentrator feed tank, pumped to the header boxes and distributed to three Falcon rougher concentrators. The header will also have a recycle back to the rougher feed tank to bypass the unit while flushing. The rougher concentrate will report to the agitated cleaner concentrator feed tank and ultrafine concentrator rougher tailings will report to the ultrafine scavenger concentrator feed tank.

Due to the very low flow rates of fresh feed to the cleaner concentrator the majority of the cleaner concentrator feed will be recycled from the header box to the feed tank to maintain high enough pumping volumes for acceptable line velocities. Rougher concentrate will be pumped to the header box feeding the cleaner concentrator. Cleaner concentrate will report to the ultrafine concentrator concentrate hopper and cleaner tailings will be recycled to the ultrafine rougher concentrator feed tank.

Similarly, to the cleaner arrangement, the majority of the scavenger concentrator feed will be recycled from the header box back to the feed tank to maintain high enough pumping volumes for acceptable line velocities. Scavenger tailings will be gravitated to the ultrafine concentrator tailings hopper and pumped to the final tailings thickener. Scavenger concentrator concentrate from the scavenger concentrator will be recycled back to the rougher concentrator feed tank.

Cleaner concentrate will be beneficiated to an acceptable grade to be combined with the coarse tin dressing concentrate to be dewatered and bagged. Due to the very low production rate in the ultrafine concentrator the circuit will predominantly recycle the concentrate back to the concentrate hopper and only when there is an acceptable level of concentrate will the pump automatically be directed to the concentrate tank.

6.6.6 Tin Dressing, Concentrate Dispatch and Tailings Disposal

Tin Dressing

The tin dressing feed material will comprise a combination of free cassiterite, free sulphides, and high to medium density composite middlings from the coarse and regrind gravity separation circuits.

The majority of the middlings particles after milling will be cassiterite bonded with silica (tourmaline or quartz) although cassiterite sulphide middlings do also occur. All cassiterite/sulphide middlings will report to tin dressing as well as some cassiterite/silica middlings depending on the tin content and particle specific gravity. Due to their composition tailings streams from tin dressing will be recycled to the regrind circuit to enable cassiterite grains to be further liberated.

The front end spiral and table beneficiation will upgrade the ROM tin grade from approximately 1% to 20% to 25% Sn. Tin dressing is needed to further beneficiate to an acceptable concentrate grade of 60% Sn and reduce the sulphur and iron impurity levels.

Due to the small tonnages within tin dressing, pumping the slurry will be very difficult as pipeline velocities must be greater than the deposition velocity to prevent settling. Dilution water will be required to aid transportation and reduce the risk of line blockages. The circuit will pump the dilute slurry to the top of the tin dressing building where it will be dewatered to the correct density and then gravity fed through the various separation units.

All tin dressing feed will originate from the gravity concentrate hopper in the spirals building. The slurry will then be pumped to the agitated tin dressing surge tank. This surge tank decouples the front end milling and gravity plants from the tin dressing circuit and allows the rest of the plant to operate for up to four hours if the tin dressing area is not operational.

The slurry will be pumped from the surge tank to the tin dressing settling cone which dewaters the tin dressing feed to acceptable densities for the downstream equipment. The settling cone overflow will be directed either to the gravity concentrate launder or the sulphides hopper using manual valves in the settling cone overflow launder. Settling cone underflow will be gravitated to the tin dressing LIMS which will remove ferromagnetic sulphides such as pyrrhotite from the concentrate. The magnetics will gravity feed to the sulphides hopper and the non-magnetics will report to the tin dressing conditioning tanks. Sulphide flotation activator, collector and pH modifier will be mixed with the slurry which will be fed via gravity to the tin dressing sulphide flotation cells. Flotation concentrate will contain mainly pyrite and arsenopyrite and will report to the sulphides hopper via the tin dressing flotation concentrate launder, while the flotation tailings containing the cassiterite will be fed by gravity to the tin dressing table feed box.

The sulphide flotation tailings will contain mostly cassiterite and some heavy middlings particles that have been misplaced in the upfront gravity circuit. A final stage of gravity separation will be required to remove the heavy middlings and produce a marketable cassiterite concentrate. The design will utilise three cleaner shaking tables and a single scavenger table to achieve high efficiency separation while maintaining adequate recoveries.

The tin dressing table feed box will distribute the slurry to three tin dressing cleaner tables. The table middlings will be collected and pumped to the tin dressing scavenger table. All dressing table tailings will be collected and pumped to the spiral middlings hopper to be reground.

All tin dressing table concentrates drop into the tin dressing table concentrate chute which will then be flushed and pumped to the dewatering area. Due to the low tonnage of tin dressing concentrate the majority of the pumped concentrate is recycled back to the concentrate chute to be used as flushing water. This will reduce the amount of water needed in the circuit maintaining a reasonable density for dewatering.

Concentrate Dispatch

Tin dressing and ultrafine processing concentrate will report to the agitated concentrate surge tank. The concentrate tank will be designed for an 18 hour residence time. The filter feed pumps will transfer the concentrate to the concentrate plate filter. Filter cake will drop on to the concentrate shuttle conveyor for bagging. Filtrate will report to the filtrate settling hopper which overflows to the filtrate hopper and will then be pumped to the mill discharge hopper.

The Achmmach bagging plant will consist of a shuttle conveyor which will transport the dewatered concentrate to two bagging units or an emergency bunker. The bagging station will consist of two online bulk bags with load cells. The change between bags will be automatic as the bag weight will trigger a high alarm and the shuttle will switch to the second bag allowing the now offline bag to be removed and replaced. If both bags are full the shuttle will move to the emergency concentrate bunker.

Concentrate containers will be transported by truck to Casablanca Port prior to consolidation and shipping to an offshore tin smelter.

Tailings Disposal

Tailings separated within the process plant will report to the tailings paste thickener feed box which will flow into the tailings paste thickener feed well. Flocculant will be mixed with the tailings slurry in the feed box and feed launder to aid settling. Thickener overflow will report to the process water tank and thickener underflow will be pumped to the tailings storage facility (TSF). A paste thickener has been selected as it is essential that the amount of water used in the plant is minimised and one key method to achieve this will be to reduce the amount of water pumped to the TSF.

As the deposited sulphide minerals in the TSF oxidise, they will generate acid which will lower the pH of the stored tailings with potential environmental implications. To neutralise the acidity, limestone will be dosed with the TSF feed.

The thickener underflow will be pumped through the tailings sampler to produce a control sample.

Plant tailings will be pumped through spigots surrounding the TSF. The particles will settle and the effluent decant water will clarify and form a decant pond. Two decant water pumps will return the clarified water to the process water tank. Section 21 of this report describes the design and operating philosophy developed by Golder for the TSF and water storage facility, which were not within the scope of the process plant work done by LADP.

6.7 Tailings Disposal

The TSF will be constructed on a phased downstream basis for the first 4 years due to the high rate of rise, then subsequently on an upstream basis.

The paste thickener underflow will be pumped along the valley contours to the TSF in a pipeline that runs around the periphery of the facility and along the constructed wall at 70% to 72% solids by weight. This will enable open spigot discharge at any point along the constructed wall to initially provide a protective solids barrier against the constructed wall, and later along the shore to enable control over the decant pond position by displacement. This will be a single point discharge at relatively high density, with the ability to extend the discharge line out from the tailings pipeline into the TSF area across the solids surface. This ultimately enables the formation of convex deposition beaches which are free draining (due to low size segregation at the elevated density), provide maximum utilisation of the TSF volume, and a natural landform for revegetation. This solids deposition will also provide a competent base formation for the later upstream wall lifts.

Limestone will be added in finely ground solid form at the eductor (inline mixer) installed in the discharge line from the tailings pump. At an equivalent of 30 kg/t tails addition rate the limestone will be sufficient to suppress the oxidation of the sulphide minerals in the tailings.

The pond water is transferred back to the process water tank using a pontoon mounted centrifugal pump.

6.8 Process Flow Summary

A high-level mass balance is presented in Table 6-14.

Table 6-14 High Level Mass balance

Annual mined tonnes	750,000 ktpa
Mechanical/electrical availability	95.0%
Process availability	95.0%
Utilisation	90.3%
Hours PA	8,760 hrs
Utilised hours	7,906 hrs

Table 6-15 is a summary of the process flow.

Table 6-15 Metallurgical performance predictions from 2018 representative sample

	tph	Ore mass (%)	Grade (%Sn)	Total Sn Rec (%)	Stage Rec (%)
Plant Feed	66.0	69.6	1.1	93.6	
Gravity Concentrate	1.5	1.5	35.0	65.5	70.0
Dressed Gravity conc	0.8	0.8	65.0	62.3	95.0
Coarse Gravity tail	18.5	19.5	0.2	5.0	
Fine Gravity Tail	5.2	5.5	0.5	3.0	
Deslime tail	11.7	12.3	0.6	9.0	
Flotation feed	35.1	37.0	0.4	17.4	
Flotation conc	0.4	0.4	29.0	15.6	90.0
Flotation tail	34.7	36.6	0.0	1.7	
UF Falcon tail	0.2	0.2	3.3	0.7	
UF Falcon concentrate	0.3	0.3	46.0	14.9	95.5
Final Combined Concentrate	**1.0**	**1.1**	**60.2**	**77.2**	**82.4**
Final tail	65.0	68.6	0.2	16.4	17.6
Total recovery from ROM ore					77.2

6.9 Process Plant Reagents

6.9.1 Flocculant

Flocculant will be used to assist with tailings thickening. The flocculant plant is expected to be purchased as a vendor package.

Flocculant powder will be received in a hopper and added to a mixing tank by means of a blower and in-line mixer. Water will be added to the tank and a flocculant suspension will be established using appropriate agitation. Mixed flocculant will be transferred to a storage tank from which it will be pumped to the tailings thickener.

6.9.2 Flotation Reagents

- MIBC frother will be received in bulk containers and transferred to an agitated holding tank in which it will be mixed and diluted with incoming water. Dosing pumps and a ring main will be used to add frother to the various flotation circuits as needed.
- H57 frother will be received in bulk containers and transferred to an agitated holding tank in which it will be mixed and diluted with incoming water. Dosing pumps and a ring main will be used to add frother to the various flotation circuits as needed.
- SIPX sulphide flotation collector will be received in bags. These will be hoisted over the mixing tank and will be discharged into the agitated tank in which SIPX will be mixed with water. Small quantities of SIPX will be dosed via a ring main to the sulphide flotation steps, with a significant internal recycle to the mixing tank.
- Concentrated sulphuric acid received in bulk containers will be dosed directly to the cassiterite flotation circuits. The great majority of the plant will reach steady state pH between 2.5 and 5 as a consequence of the use of sulphuric acid in the flotation circuit. Materials of construction have been specified accordingly.
- SSF sulphide flotation modifier will be received in bags. These will be hoisted over the mixing tank and will be discharged into the agitated tank in which SSF will be mixed with water. Small quantities of SSF will be dosed via a ring main to the sulphide flotation steps, with a significant internal recycle to the mixing tank.
- SPA and OPS30 cassiterite flotation collectors will be received in bags and are mixed in a 50:50 ratio. These will be hoisted over the mixing tank and will be discharged into the agitated tank in which SPA/OPS30 will be mixed with water. Small quantities will be dosed via a ring main to the cassiterite flotation steps, with a significant internal recycle to the mixing tank.
- Limestone will be used to neutralise tailings before pumping the material to the TSF. Crushed limestone will be received in trucks at the limestone stockpile and will be reclaimed and conveyed to a small limestone mill. Milled limestone will be discharged into a hopper and pumped to an agitated surge tank. Centrifugal pumps will then be used to transfer limestone slurry to the tailings transfer tank.
- FeSS is a cassiterite flotation depressant and is made on site by mixing $FeSO_4$ and SS, both these reagents being delivered in bags. Small quantities of FeSS will be dosed via a ring main to the cassiterite flotation steps, with a significant internal recycle to the mixing tank.
- Acidified Sodium Silicate is an activator used in the sulphide flotation steps. This is also made on site by mixing Sodium Silicate and Sulphuric acid and is dosed to the flotation area.

6.9.3 Reagent Supply

Reagents are supplied as powder or prill in bags or as liquids in bulki-boxes. Some of the liquids are added to the process with dosing pumps at as received strength, either directly from the bulki-box or after transfer to a dosing tank. The powder prill reagents are transferred in a known mass to a mixing tank where a pre-calculated amount of water is added to a level in the mix tank. Once the mix is complete the solution is transferred to a dosing tank.

The solutions are then pumped in a pressured ring main around the selected process area and solution is tapped from the ring main at selected locations into the process via solenoid valves controlled by timers.

6.10 Samplers

Sampling will utilise the actuated and static in pipe cutter systems as typically supplied by Thermo-Gammametrics. On small flow rates the entire stream is diverted pneumatically to a cutter box at selected time intervals, and on larger streams a two-stage system using an in-pipe pressure sampler or cutter which provide a 1% to 5% of total flow sample to a secondary pneumatic cutter.

6.11 Water Services

Water is supplied from the water storage facility (WSF) into a raw water tank. This provides makeup water to the process water tank, supplies specific areas in the plant directly such as safety showers, spray bars and reagents mixing, and provides feed to the potable water system. Half of the volume of the raw water tank is reserved to be available as fire water supply.

6.12 Air Services

Air Compressors are installed for plant and instrument air requirements throughout the plant.

6.13 Process Control System

The control system will be a basic PLC control system with the facility to drive PID control loops such as pump speed to level control measurement and water addition to flowrate measurements, sometimes with a simple cascade such as from a density measurement and control block.

Basic algorithms to calculate a setpoint or modify a process variable; for example, the difference between two weightometer measurements-will be implementable

7 WATER & TAILINGS STORAGE FACILITIES

7.1 Tailings Storage Facility (TSF)

7.1.1 Background

Golder was engaged to prepare all requirements related to tailings storage design and management for the project. The scope of work included a geotechnical investigation and laboratory test work for determining the engineering properties of the construction materials and tailings, assistance with the interpretation of geophysical test results, preparation of conceptual designs based on tailings and water storage facility (WSF) requirements for the TSF and WSF respectively.

The Golder report is included in Appendix 7A.

7.1.2 Proposed Location

The TSF will be located approximately 300 m to the south west of the processing plant in a predominately non-forest area. The Achmmach Tin project will install ore sorting technology at the front of the processing plant. Achmmach will produce 7 million tonnes of ore over the life of the mine. This ore will be crushed and passed through an ore sorter. The ore sorter will reject the waste component from mined ore and will deliver 4.9 million tonnes of mill feed to the processing facility. The TSF will accommodate all 4.9 million tonnes of the processed ore.

7.1.3 Site Conditions and Investigation

Field assessments of the proposed TSF location were completed by Golder in 2017 and included site mapping, geotechnical trial pitting, overburden logging, rotary core drill holes.

From the different range of laboratory tests performed, the results show that the geotechnical properties of the soils encountered are all very similar. The only major difference noted from the properties of the soils is the amount of clay content is much greater for the slump material, soil type 2, found on the northern slope within the TSF footprint.

The rock encountered in the drilling program is weak sericite altered mudstone. The tested uniaxial strength ranged from 1.8 MPa to 5.3 MPa. This uniaxial strength will not be exceeded by the load implied by the construction of the TSF embankment. Slake durability testing indicates that the in-situ mudstones are sufficiently durable to suggest they can be used as embankment construction fill. Care will be needed when excavating, screening, transporting, placing and compacting the mudstone to prevent it being de-structured, as this could potentially lead to the formation of a lower permeability embankment opposed to the free draining rock fill embankments proposed. Based on the results of the geotechnical results and Golder's experience with similar materials effective shear strength parameters of $c' = 5$ kPa and $\phi' = 33°$ have been adopted for this study.

7.1.4 Design

The proposed TSF is a valley impoundment and construction main embankment (starter wall) across the valley to the south west of the process plant. The tailings in the facility will be contained along the southern, eastern and northern edges of the facility by the natural contours of the valley, and on the western side by the starter wall and future wall raises to be constructed above the wall. The initial starter embankment will be constructed to an elevation of 930.5 masl (thus 23 m high measured from the final outer toe of the wall at approximately 907.5 masl, to its crest). This will be followed by the construction of three downstream raises to elevations of 937.5 masl, 942.5 masl and 944.5 masl, for a final starter wall height of 37 m). A further series of five 3m high upstream raises will take the embankment to a final elevation of 959.5 masl, thus a maximum final height of 52 m at closure of the facility.

The facility will have a storage capacity of approximately 5.55 Mm^3 at the closure elevation of 959.5 masl. At closure the TSF will have a top tailings surface area of approximately 194,000 m^2. Table 7-1 shows the TSF embankment stages.

Table 7-1 Embankment stages

Stage	Wall Material (m^3)	Dam Crest (masl)	Construction Method	Months of Storage	ROR (m/year)
1	107,226	930.5	downstream	12	10
2	139,473	937.5	downstream	13	8
3	140,920	942.5	downstream	13	4
4	69,327	944.5	downstream	8	3
5	12,784	947.5	upstream	13	2.8
6	13,512	950.5	upstream	16	2.5
7	14,145	953.5	upstream	17	2.2
8	14,986	956.5	upstream	18	2.0
9	15,897	959.5	upstream	19	1.9

In order to limit the quantity of material required for the embankment raises, upstream rock fill embankment raises will be incorporated into the construction of the TSF. The construction of the TSF starter embankment will be carried out in four phases, the first to 930.5 masl requiring approximately 107,226 m3 of rock fill, with three downstream raises to 937.5 masl, 942.5 masl and 944.5 masl, requiring a further 139,473 m^3, 140,920 m^3 and 69,327 m^3 of rock fill respectively. The first four stages of construction carried in operational years -1, 1, 2 and 3, will form the starter embankment, and will provide approximately 3.83 years of storage capacity.

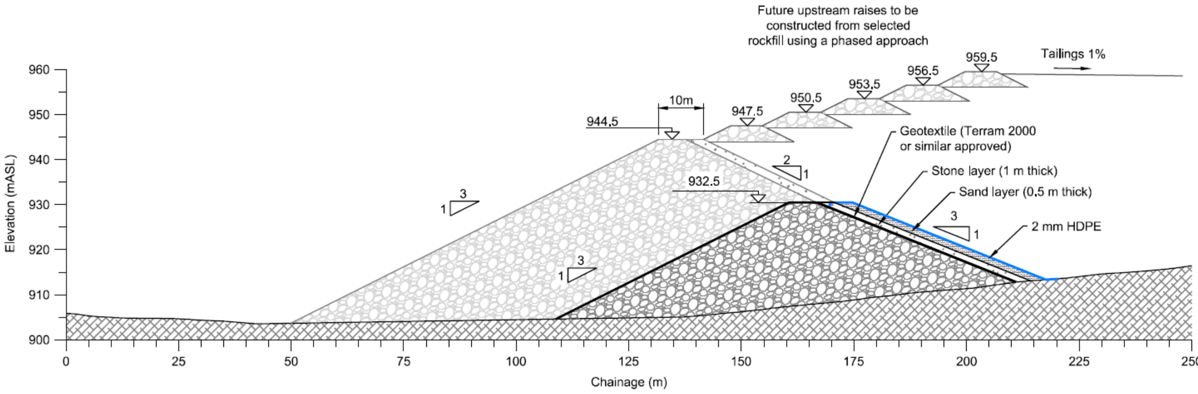


Figure 7-1 TSF typical embankment section

The subsequent upstream raises can only start when the rate of rise decreases to 3 m or less per year. The rate of rise decreases from 4 mpa to 3 mpa between elevations 942.5 masl and 944.5 masl. From this time on the TSF will be developed through the construction of a five 3 m high upstream rockfill raises until the proposed closure elevation of 959.5 masl is reached. Construction of the upstream raises will require between approximately 12,000 m^3 and 16,000 m^3 of rockfill, with stage construction required every 12 months.

The phased starter embankment will have upstream and downstream slopes of 3H:1V and a crest width of 6 m. Subsequent upstream wall raises will be have an upstream and downstream slopes of 2H:1V with a crest width of 5 m for an overall slope of approximately 4H:1V.

The TSF was analysed at a throughput rate of 750 ktpa for:

- Failure of the downstream face (sloughing) for both the starter dam and final profile using a grid and radius slip surface and utilising the Morgenstern-Price method; and
- Global failure of the downstream face for both the starter dam and final profile using an entry and exit slip surface and utilising the Morgenstern-Price method.

The slope analyses were carried out for both static and pseudo-static conditions (to assess the impact of earthquake loading). Analyses were carried out based on an estimated normal operating piezometric surface and elevated piezometric surface.

A Factor of Safety (FOS) of 1.50 is considered satisfactory for the long term static loading and a FOS of 1 is considered satisfactory for pseudo-static conditions for a seismic assessment, with a FOS of 1.2 to 1.3 considered acceptable for post-earthquake conditions. The analyses carried out as part of the 2017 DFS Update were carried out on a 37.5 m high, 3H:1V starter embankment with a final height of 55.0 m and a 4.5H:1V average slope.

The results suggested that the overall downstream embankment, for the starter embankment and at full height facility will remain stable in both the static and pseudo-static (seismic loading) conditions, as long as the phreatic surface within the facility is controlled. In the event that the phreatic surface in the tailings is not controlled, the TSF would require modification with a buttress or flatter slopes to meet safety requirements. It is important note that monitoring and design verification is key to the implementation of upstream raising on a TSF.

Due to the decrease in the Plant throughput to 520 ktpa, and the subsequent remodelling of the TSF walls and 3m lower proposed closure height, Golder has to update the stability assessments for the TSF to take these changes into account. This work will be carried out as part of the detailed design of the facility.

The TSF basin will be covered with clay in areas considered susceptible to seepage, and equipped with a seepage collection system. The combination of these features as well as drainage medium within the starter embankment produce a low phreatic head in the tailings mass for maintaining stability as well as mitigating against uncontrolled seepage from the facility.

A contour drain cut above the upstream extremities of the basin will provide for run-off diversion during rainfall events and snow-melt.

7.1.5 Operating Philosophy

The operating phase of the TSF will commence with commissioning of the plant and first deposition of tailings on the facility. Tailings from the process plant will be pumped from the plant to the TSF via an HDPE slurry delivery pipeline and deposited through a series of discharge spigots around the TSF.

Tailings will normally be deposited at a slurry density of 72% w/w. Initial tailings deposition will be carried out via spigots from the starter wall in an easterly direction to create a beach with a 1% fall towards the rear of the facility. This approach will push the supernatant pool from its initial position against the starter wall up-valley and towards the rear of the facility, until it reaches the toe of the WSF. During later years, tailings deposition will be carried out from the western wall raises, as well as from the southern and eastern sides of the facility, the latter to reposition the pool closer to the front of the TSF for closure purposes.

A pump barge will follow the migration of the pool to facilitate adequate return of supernatant water to the plant, as process water. The barge will be equipped with duty and standby pumps, feeding a return water pipeline to the process water tank at the plant. Supernatant water on the TSF will consist of process water from the tailings slurry and rain water direct fall and catchment run-off. A minimum pool volume of 10 000 m3 will be maintained to meet pump draft requirements.

When the pool has reached it long term position, the tailings deposition system will be extended to ensure that the tailings deposition capacity of the facility is maximised and that the pool will remain in its long term position near the plant. Additional piping will also be required for the re-positioning of the pool during the last three years of operation to prepare the facility for closure.

7.1.6 Tailings Characterisation

A screening level geochemical characterisation programme was carried out on both ore and waste rock in the proposed underground mine areas as well as on the tailings material which would be generated through processing the ore. Appendix 7B provides the Golder geochemical assessment report.

Ten drill core samples were collected by Atlas Tin from exploration boreholes drilled as part of the Resource Definition stage of the project. In addition, one tailings sample was made available for analyses. Atlas Tin also collected limestone from quarries in the vicinity of the proposed mine and blended this with the tailings to generate composite samples which were also analysed. Static and kinetic geochemical testing has been carried out on the samples. The static testing included determination of chemical composition, mineralogy, acid base accounting, NAG pH determinations and short-term leach testing (including shake flask extraction and Net Acid Generation leaching).

The following conclusions were drawn from this study:

- In general, the samples were characterised by high sulphide content and acidification potential and low buffering capacity, and therefore they are expected to have a significant overall potential to generate acid rock drainage;
- Metal mobility is a potential environmental hazard which will be considered in the final design of the tailings storage and waste rock facilities. It is important to note that metal leaching can take place under both acidic and alkaline conditions;
- Based on the SFE and NAG leaching test results, constituents of potential concern for both the short and long-term include: sulphate, arsenic, cadmium, copper, iron, manganese, nickel and zinc; and

- Addition of limestone to the tailings and waste rock results in a significant reduction in acid generation and metal leaching potential.

Little distinction could be made between the anticipated geochemical behaviour of the ore and waste rock from the samples provided for testing. Most of this material could cause ore stockpiles and waste rock dumps to turn acidic (given the low buffering capacity). Therefore, Golder recommended long-term management of this material and limited use in construction. Blending of the tailings sample with limestone has resulted in significant reduction in the acid generation and metal leaching potential.

7.2 Water Storage Facility

The WSF will provide the bulk of the project water requirements. It will be located directly upstream of the proposed TSF where a cross valley, earthen embankment will be constructed. Based on the water balance results this embankment will be constructed, in a single phase, to a final elevation of approximately 970 masl. This embankment will form a facility which will have the capacity to store approximately 330,000 m^3 of surface water runoff for use in the mine operations. To prevent excessive seepage of stored water from the facility a compacted clay liner (CCL), with an overall permeability of approximately 1x10-8 m/s, will be installed on the base, side slopes and upstream face of the WSF. The installation of the CCL will require the engineered placement of approximately 58,000 m3 of clay material.

The WSF will be equipped with an operational spillway which will tie into the storm water diversion. This will run from the north western side of the facility, past the northern side of the TSF, the wall raises and the starter wall, down to a dissipater structure from where the water will flow down the valley.

A pontoon pump will transfer water to the raw water tank in the processing plant. A second pontoon pump will be utilised to service the paste plant and mine by pumping to a central mine water header tank.

7.3 Closure Plan

Closure planning has been undertaken to a conceptual level for key infrastructure such as the tailing management facility and will be continually updated throughout the Project life. Prior to the start of production activities, a conceptual mine rehabilitation and closure plan (MRCP) will be prepared.

The preparation of the detailed MRCP will be an iterative process that will evolve over the life of the Project taking into consideration views and concerns of the local communities and monitoring information. The MRCP will be finalised at least three years prior to the end of the mines operating life. Once the mine has finished producing, the detailed closure plan will be implemented.

Provision for closure and rehabilitation of the site will be made during the operation of the mine. The following key activities will form the basis of the rehabilitation and decommissioning process:

- Securing or blocking mine entry and exit points. This may be achieved by collapsing mine entry points or by installing secure barriers to entry.
- Development of post-mining land use and closure objectives and criteria. This may be achieved through review of initial land use practice and assessment of potential future land use in consultation with likely land users of the future.

- Design and construction of final landforms, drainage structures and any new infrastructure required. This would require commitment of machinery and resources to implement final plans.
- Estimating, reconciling and scheduling of rehabilitation material inventories. This would be an on-going process to maintain an understanding of the likely availability of materials required for effective rehabilitation and the sources of these materials.
- Demolition and decommissioning of plant and infrastructure. Residual material may be sealed in the underground workings.
- Landform surface treatments (ripping, selective application of topsoil, placement of materials). This work would be carried out to establish opportunity for moisture retention and effective re-vegetation.
- Completion of rehabilitation (including revegetation). Rehabilitation plans would be executed by employing appropriate local personnel to carry out land preparation and specific planting and husbandry programmes.
- Commence monitoring and measurement against completion criteria. – On-going consultation with key stakeholders.
- Handover of infrastructure and finalise post closure monitoring and maintenance programme.

Although the Projects closure plan is yet to be produced, a number of environmental and social concerns have been identified during stakeholder consultation. These being:

- Community Health and safety: The physical presence of the buildings, waste rock dumps, tailings and process equipment are of concern in terms of risk to community health and safety.
- Risk of pollution of water resources: The TSF could represent a source of acid leaching.
- Erosion / instability: There is concern with regards to the stability of waste rock dumps and the TSF, especially with respect to run-off water, sedimentation and risk of seismic activity.

All these items are matters that are typically managed during operations.

7.4 Storm Water Diversion

The TSF and the WSF are located in a valley with a surrounding catchment area, the majority of which is located to the north and east of the facility. A storm water management system will aim to separate clean water to be stored in the WSF from process water stored on the TSF. Golder estimates it will be possible to collect and store water during the wetter winter months to supply the raw water needs of the process throughout the entire year. Storm water from the majority of the catchment area will be captured in the WSF. With the challenging topography and tourmaline outcrops to the south of the WSF and TSF, it would also be difficult to construct a storm water diversion around this side of the facilities, and run-off from this area will report to both the WSF and TSF. Figure 7-2 illustrates the catchment strategy for the TSF and WSF.

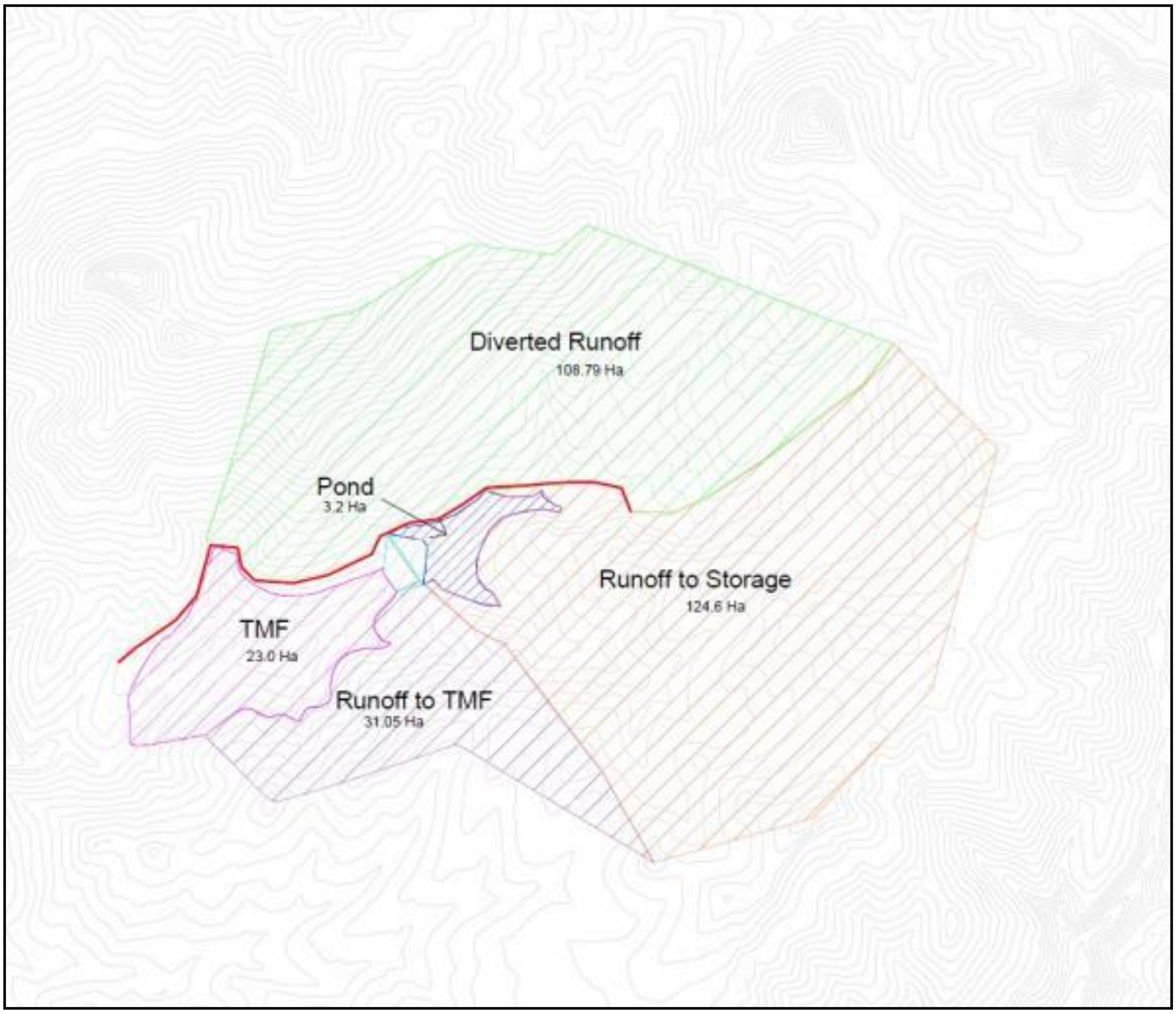


Figure 7-2 TSF & WSF run-off catchment

The WSF will be equipped with an operational spillway which will tie into the storm water diversion. This will run from the north western side of the facility, past the northern side of the TSF, the wall raises and the starter wall, down to a dissipater structure from where the water will flow down the valley.

Storm water on the TSF will be managed by diverting most of the water from the catchment to the north of the facility around the TSF and down to the valley below on the western side. Storm water from the catchment area to the south of the facility will be managed on the TSF. The TSF will be equipped with an emergency decant system to prevent the facility from over topping should a large storm event occur after commissioning. Water from the underground operations will also be pumped to the TSF for clarification, before being pumped into the process water system which will in turn supply the mine. For closure of the facility, it will also be equipped with a spillway system which will tie into the storm water diversion system to be constructed to the north of the facility.

7.5 TSF and WSF costs estimates

The costs for the WSF allow for it to be constructed before the TSF, as is planned by Atlas Tin to allow maximum water harvesting prior to plant commissioning. All costs are summarised in section 14.

8 OPERATIONS MANAGEMENT

8.1.1 Introduction

The Project is located 55 km from the major regional centre of Meknès (pop ~ 600,000), and 150 km from Rabat the capital of Morocco (pop ~ 1.2 million). The region is an area of historic and ongoing underground mining operations. Morocco hosts an established modern mining industry and associated support services industries.

Atlas Tin proposes to maintain an administrative head office in Meknès, close to commercial, business and government department and services. This administration office will house the key functions of finance, commercial, human resources, external relations and legal compliance.

Facilities will be established at the Achmmach mine site to support all production activities, this includes, mining, processing, maintenance, safety, environmental, stores, IT and logistics. There is an existing accommodation camp at the project site that will be increased to 55 bed capacity and will accommodate expatriate and essential site based Moroccan employees. It is anticipated that the workforce will be bussed in daily and/or travel in from the local communities. Bus transportation from Meknès will be provided by Atlas Tin.

The project will directly employ approximately 160 people and site contractors will employ a further 178 people during the peak of production. Employees will be recruited locally and within the wider Moroccan region, and the key expatriates are expected to be sourced from Europe and Australia. It is likely that all Moroccan labour will be employed out of Meknès. The labour breakdown is summarised in Table 8-1 below.

Table 8-1 Labour break down

Department	National	Expatriate	Total
Management	15	5	20
Site Admin	35	1	36
Processing	69	4	73
Mining Atlas	29	4	33
Mining Contractor	147	31	178
Total Workforce	295	45	340

Atlas Tin has developed detailed organisational charts to support a progressive recruitment schedule to allow the operating team to transition from construction and into operations. This is planned to occur allowing adequate time for training in operational policies and procedures to ensure the successful ramp up and development of the project. A key component to the progressive recruitment philosophy is the employment of expatriate technical and training personnel in the early stage of the project construction to support the development of policies procedures and the training of the Moroccan workforce. These expatriate personnel will transition out as the systems, policies and procedures are embedded. It is envisaged that by the time the mine reaches full production the workforce will greater than 90% Moroccan.

Detailed organisation structures have been developed to support the feasibility and the understanding of the required skills, training and capabilities of the Moroccan workforce.

8.1.2 Labour Survey

A social baseline survey was conducted in 2014 as part of the ESIA (environmental and social impact assessment); discussed in Section 10. The aim of the survey was to establish the current social, economic and cultural context of the project area. Fieldwork consisted of a household survey, focus group discussions with representatives of the communities adjacent to the Project and semi-structured interviews with social workers in the fields of health, local development and education and representatives of local authorities.

The Achmmach Project is located within the administrative region of Meknès-Tafilalet, in the Caidad of Jahjouh, Cercle of Agourai, El Hajeb Province. While the Concession area lies within two rural communes, Ras Ijerri and Ait Ouikhalfen, the direct area of influence of the project includes a third commune, close to the concession, Jahjouh. During the consultation process, local authorities insisted on this definition of the direct sphere of influence of the project (i.e. the three communes), particularly in relation to their interpretation of local content and job expectations at the local level. The total population of the three rural districts in which the project is located is estimated at 19,700 people. Ras Ijerri and Sebt Jahjouh are the two principal urban centres which concentrate the majority of the population. The project area is part of the area inhabited by a group of people collectively known as the Guerrouane du Sud. While Berber is the language of the majority of the population (the Guerrouane du Sud are originally of Berber origin), there are also many Arabic and French speakers.

Within the concession area there are 50 households representing an estimated 300 people. Much of the rural population around the project lives under or close to the poverty line. The project area registers very low adult literacy rate.

Agriculture is the primary activity and source of income. Main cultivated crops include wheat, barley, oats, broad beans and olives; livestock consists of poultry, goats, sheep, cattle, horses, mules and donkeys.

The project can contribute to a general improvement of quality of life and health in the area. The positive social impacts include:

- creation of direct employment during the construction phase and mining operations over at least 11 years
- payment of rent for the use of the collective lands. This will be a constant source of income and may help offset seasons of poor yields
- job opportunities for young people who can remain in the district rather than to leave the district to seek work elsewhere.

8.1.3 Recruitment and Employment Action Plan

As part of the pre- production activities Atlas Tin will develop an employee action plan. The site general manager will have overall accountability for the employee action plan however both the human resources manager and the community liaison officer will be tasked with the day to day implementation and management of action plan.

In accordance with the ESIA, the objective of the action plan is to maximise local employment. Job vacancies will be advertised locally in the first instance and jobs which are not be filled by local applicants will be advertised on a national scale. In addition, Atlas Tin will establish:

- a recruitment office in one of the local towns

- a grievance mechanism (detailed in Section 10.7)
- regular communication with local authorities to discuss any concerns that have been expressed by the local people
- a code of conduct and training for employees and contractors that sets out how to respect the local people and their customs.

A key component to the progressive recruitment philosophy is the recruitment of expatriate technical and training personnel in the early stage of the project to support the development of policies procedures and the training of the Moroccan workforce. These expatriate personnel will transition out as the systems, skills, policies and procedures are embedded. It is envisaged that during the peak production the Achmmach project will have a workforce that is at least 90% Moroccan.

Existing Employees

Atlas Tin currently has nine employees, all of whom were employed from Meknès and surrounds. It is intended that these people will continue to be employed as the operation moves in to production. Atlas Tin has also employed local people in casual positions as required from time to time.

Labour Needs Analysis

The labour requirements for the project are summarised in Table 8-2 below. Labour needs have been defined in four broad categories to identify the key level of education training and skills required for the project.

Level 1 – Tertiary / diploma level education

Level 2 – Diploma trade or certificate qualifications or specialised experience or training.

Level 3 – Secondary level of education and literacy and /or experience in industry.

Level 4 – Includes employees with limited literacy skills and experience

Table 8-2 Moroccan labour requirements analysis

	Level 1	Level 2	Level 3	Level 4
Management	7	8		
Site Administration	16	3	3	13
Processing	10	25	27	7
Mining	11	8	6	4
Mining Contractor	2	31	104	10
Total	**46**	**75**	**140**	**34**

These numbers have been developed for the feasibility study and may vary during the implementation and operational phase. The training needs analysis has indicated that the local communities and surrounding region, including Meknès will be able to support a large portion of the employment requirements of the project.

8.1.4 Moroccan Labour Laws

Morocco has established labour laws and practices, including practices specific to mining operations. Atlas Tin has maintained ongoing local employment for a period of 10 years and has a working knowledge of employer-employee laws and obligations. These are summarised below:

- law no.65-99 of 11 September 2003 provides for the labour code
- decree of 24 December 1960 on the status of mining company personnel, as amended and supplemented
- decree of 23 March 1993 setting out the incentives for enterprises which provide training and occupational integration as amended by Law 39-06
- decree no.2-04-469 of 26 December 2004 on the notice for unilateral termination of the unlimited duration labour contract
- ministerial order no. 350-05 of 9 February 2005 establishing the standard work agreement for foreigners (CTE) and the documents required for the application to obtain a CTE visa
- ministerial order no. 1391-05 of 25 November 2005 fixing the list of categories of persons exempt from the ANAPEC certificate of activity (certificat d'activité)
- 2018 Guide on the employment of foreign employees in Morocco, Ministry of Employment and Vocational Training

Types of employment contracts

The different types of employment contracts and their conditions are set out in Table 8-3 below:

Table 8-3 Moroccan employment contracts

CONTRACT	CONDITIONS
Fixed-term contract (CDD)	For a period of one year, renewable once. Where the CDD is extended further, it automatically becomes a permanent contract (CDI).
Permanent contract (CDI)	May include a trial period of: 3 months, renewable once, for management positions (cadre); 1.5 months, renewable once, for non-management employees; and 15 days, renewable once, for workers.
The professional integration (traineeship) contract (CIP), registered with the National agency for the promotion of employment and working skills (ANAPEC)	The employer is exempt from paying employer's social contributions and from the professional training tax and the trainees are exempted from the payment of the income tax on their traineeship grant. Has a 24-month term, renewable once for an additional period of 12 months. Remuneration must be between MAD 1,600 (US$ 170) and MAD 6,000 (US$ 660). Where the remuneration exceeds MAD 6,000 (US$ 660), the incentives described above are not available.

Atlas Tin will utilise all three categories of employment.

Employer Obligations

Key obligations of the employer are detailed in the Labour Code (articles 21 to 24). These include requirements to:

- provide employees with a work permit (carte de travail)
- protect the health, safety and dignity of employees in the workplace
- provide employees with the internal regulations of the company; relevant collective employment agreements (conventions collectives de travail); work schedule; procedures in relation to weekly rest periods; legal provisions relating to health and safety and prevention; and the date, time and place of payment
- provide employees with the relevant registration number with the National Social Security Fund (CNSS)

- provide employees with the name of the insurance organisation covering work-related accidents and occupational diseases
- maintain a social security and tax register, and register all employees and trainees, with the CNSS (labour code, article 24)
- deduct income tax at source on behalf of the Moroccan tax administration (general tax code, articles 56 et seq).

In addition to the key employee obligations summarized above the Moroccan labour laws outline further provisions for redundancy, termination and labour laws specific to the employment of mining personnel.

Rules Governing Mining Personnel

The decree of 24 December 1960 on the status of mining company personnel ("Mining Personnel Decree") applies to mining company with more than 300 employees and by order of the Minister of Mines, it may also be applied to mining companies with more than 100 employees. It sets out a separate regime applicable to mining company personnel, with the general regime continuing to apply where the Mining Personnel Decree is silent. Key provisions include:

- For technicians, supervisors and administrative managerial (cadre) personnel the trial period is a fixed non-renewable period of 12 working days for workers and non-management employees and one month.
- For workers and employees the trial period is three-months after which they will be considered permanent, except for those working at research sites (chantiers de recherche) of the company, in which case the period is one year of continuous employment.
- For supervisors, technicians and management recruited for such positions or those holding a qualification from a technical or vocational school or apprenticeship centre, as well as those recruited from among the workers or non-management employees, are subject to a six-month internship, at the end of which they are tenured on the recommendation of the relevant head of department. If the individual is not appointed, then they stay in their previous position.

Permanent personnel under the Mining Personnel Decree may be dismissed in the following situations (in addition to the circumstances under the general regime described in section 7.4 above):

- collective dismissal
- physical incapacity after a medical examination
- professional incompetence following a warning letter sent to the employee and the staff and internal regulations commission has investigated the matter.

Permanent personnel may resign by providing 12 working days' notice for workers and non-management employees or one months' notice for technicians, supervisors and administrative, managerial (cadre) personnel.

The Mining Personnel Decree includes an obligation on mining companies to establish a staff and internal regulations commission. The objectives of the commission are to:

- oversee the application of the internal regulations
- review personnel claims relating to recruitment, tenure and progression, dismissal and disciplinary sanctions
- settle collective disputes.

The commission consists of between 8 and 16 members, with an equal number of representatives from among the employees and management (with terms of appointment being two years and six years, respectively).

The Mining Personnel Decree also includes an obligation to establish an advisory committee, which is required to:

- be informed about the state of the business, including work programs, production and productivity
- submit to the decision-making bodies any proposal to improve the company's performance and productivity.

Foreign employees

In Morocco, the recruitment of a foreign national is subject to the grant of an employment contract authorisation (visa), as provided under article 517 of the labour code. The employment contract must be drafted in accordance with the standard form contract for foreign employees in Morocco. Decree No. 350-05 sets out the standard form contract for foreign employees and the required documents for the application for foreign employment authorisation.

The recruitment of foreign employees is subject to a procedure whereby the employer must demonstrate that the position required specific skills that are not commonly found in Morocco and that there are no Moroccan candidates with a similar profile.

8.1.5 Organisational Structure

Detailed organisational structures have been developed to support the project to ensure the training and development, roles responsibilities and the necessary supervision is in place to ensure compliance to all Moroccan labour laws, company policies and to maintain a safe and modern mining operation.

Organisation structures have been developed to reflect the reporting structures and required to maintain key inter and intra department disciplines and coordination. The organisation charts and individual manning requirements are outlined in the following sections.

Management & Administration Organisation Structure

The management and administration team will be responsible for general oversight of the mine including the commercial, human resources, legal compliance, community and general administrative functions and also includes the key management roles for construction, mining and processing. A small head office will be established in the business centre of Meknès, where key commercial and government functions are located. All day to day operations and site management will be based at the mine site.

Whilst the general manager will be responsible for ensuring the key development and maintenance of operating policies and action plans the key leadership team will be responsible for the day to day management and implementation of them. The proposed organisation structure is shown in Figure 8-1 below.

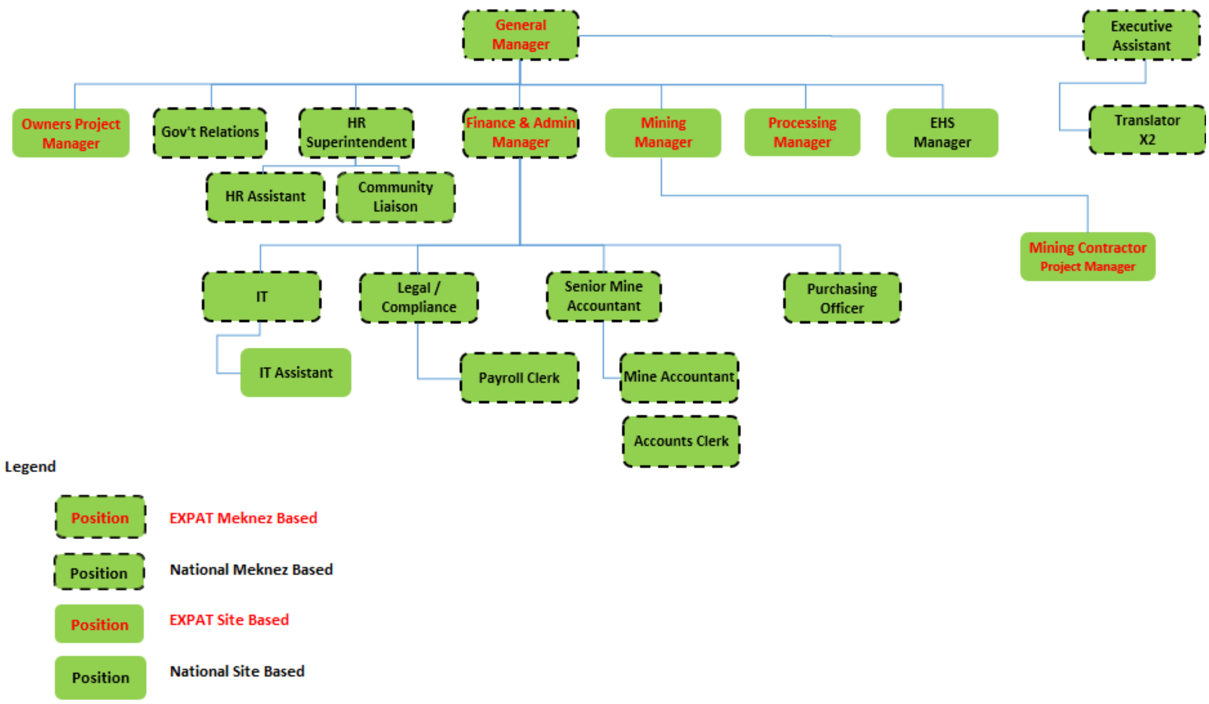


Figure 8-1 Management & administration organisation structure

Site Administration Organisational Structure

The site administration department (Figure 8-2) will be responsible for coordinating all site services including the camp, transport, logistics stores, site security as well as camp health and site medical services. The site administration team will maintain a cross functional reporting relationship with the key site management and the management and administration team based in Meknès to ensure the efficient for day to day planning and coordination of the site requirements.

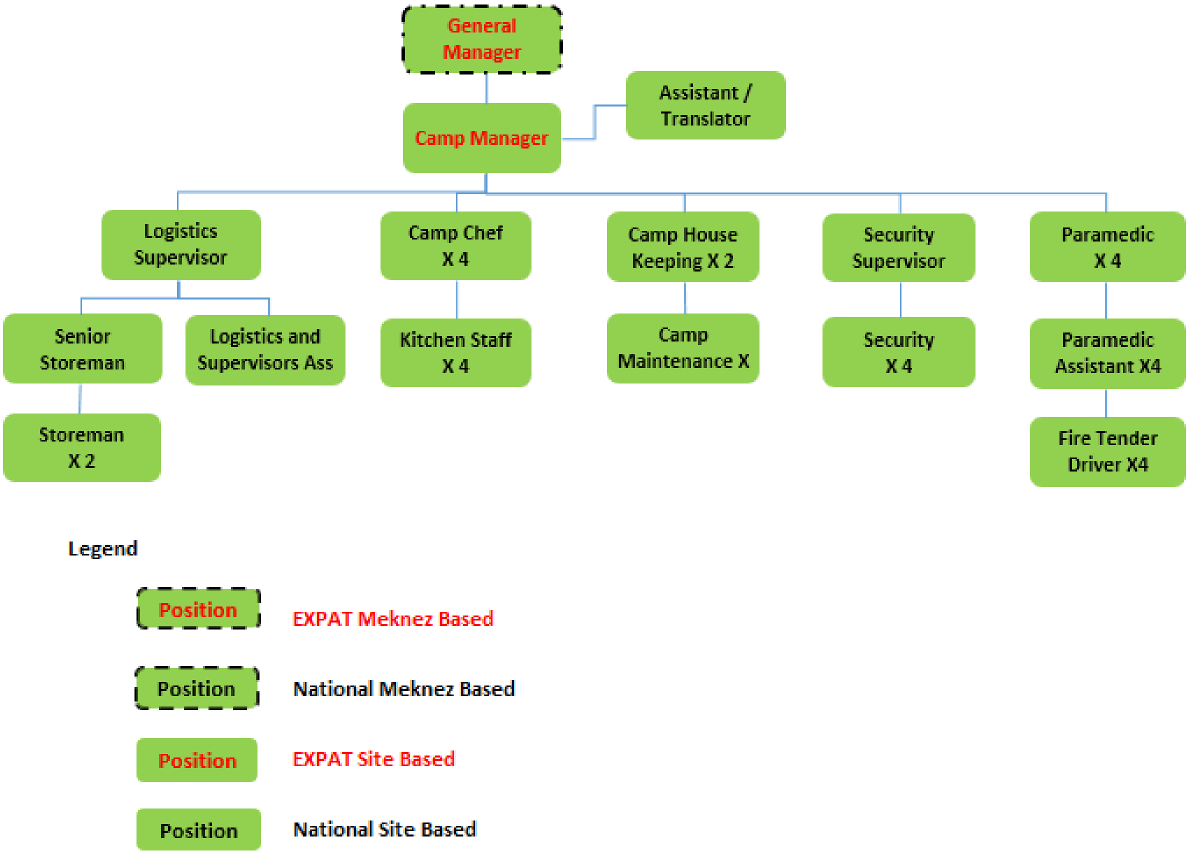


Figure 8-2 Site administration organisational structure

Processing Organisational Structure

Atlas Tin will manage and operate the processing facility and all maintenance activities laboratory as well as oversee the laboratory contractor. The site processing manager will also be responsible for the environmental action plan and overall site environmental management plan. The organisational structure is shown in Figure 8-3.

It is expected that the main maintenance shut-down activities will be undertaken by Atlas Tin employees supplemented with local and regional contractors. Morocco hosts a significant mining support industry for Moroccan mines providing the specialist skills and capability not typically employed at the mine site.

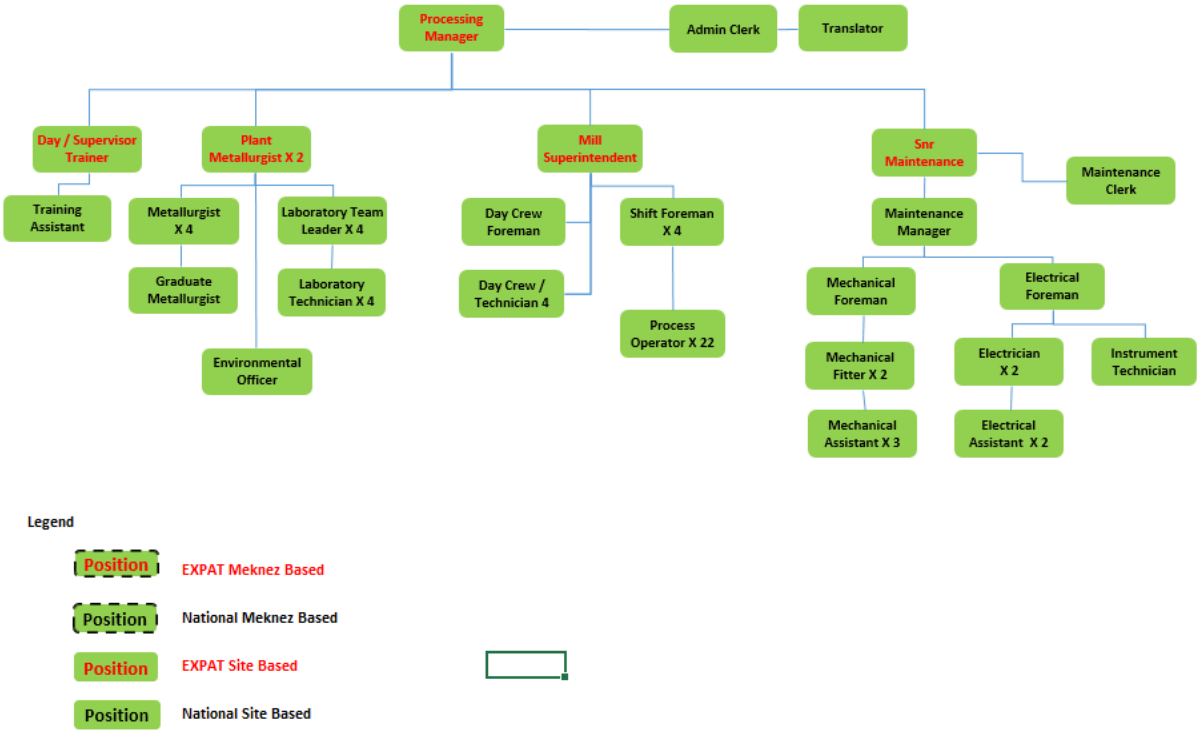


Figure 8-3 Processing organisational structure

Mining Organisational Structure

Atlas Tin will provide planning and oversight personnel for the mine (Figure 8-4), which will be operated by contractors. Atlas Tin's management and technical team will provide all the technical design, scheduling and coordination of the mining activities for the contractor. A suitably qualified contractor will be engaged to undertake mining operations.

The mining technical team includes expatriate labour to support the initial development and training of the Moroccan workforce to the standards required of a modern efficient mining operation.

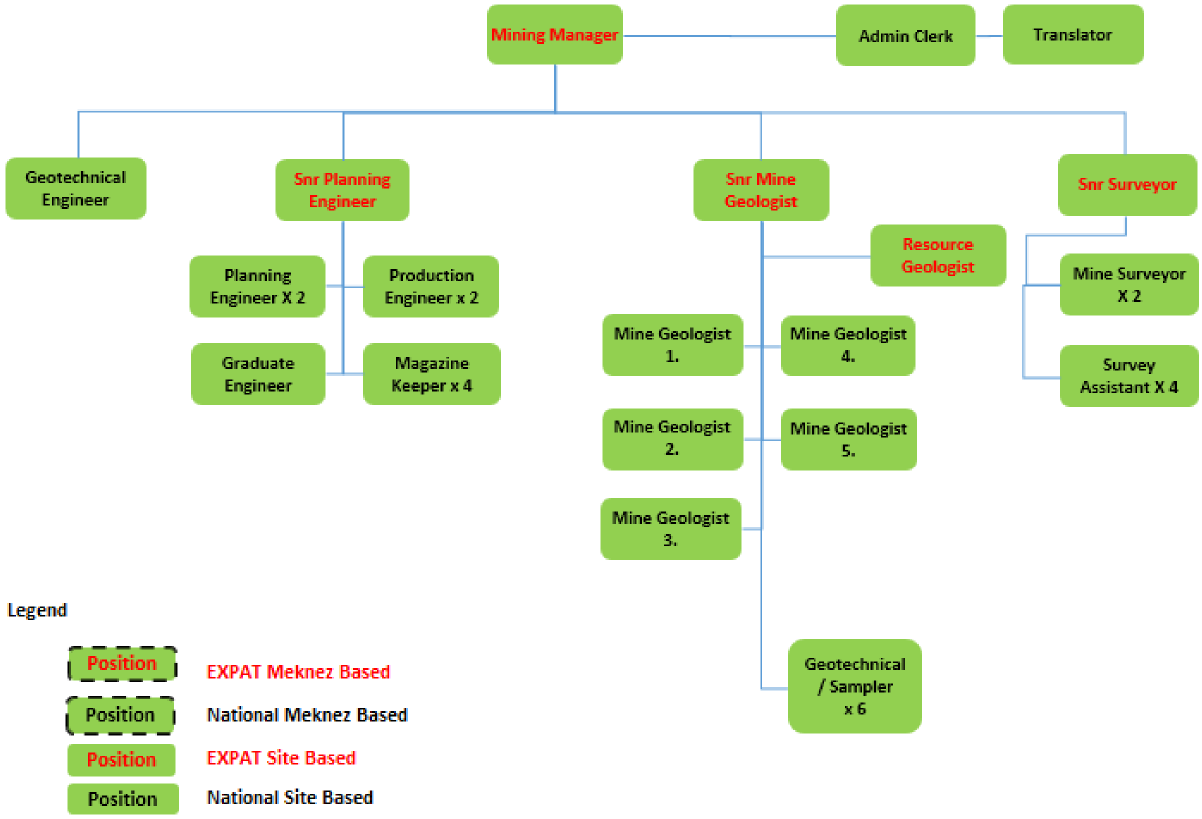


Figure 8-4 Mining organisational structure

Contractor Mining Organisational Structure

Atlas Tin has developed an understanding of the skills and capabilities of several underground mining contractors and support services globally. The organisation structure provided in this feasibility study is based on the review of pricing submissions by mining contractors and as such represents a typical mining contractor organisational structure (Figure 8-5).

Atlas Tin has sought pricing schedules from several mining contractors and is continuing to build its understanding of their capability and capacity. The selection of the preferred contractor will be undertaken during the detailed design and construction phase of the project and will be based on a selection matrix that includes capability, price, cultural and language criteria.

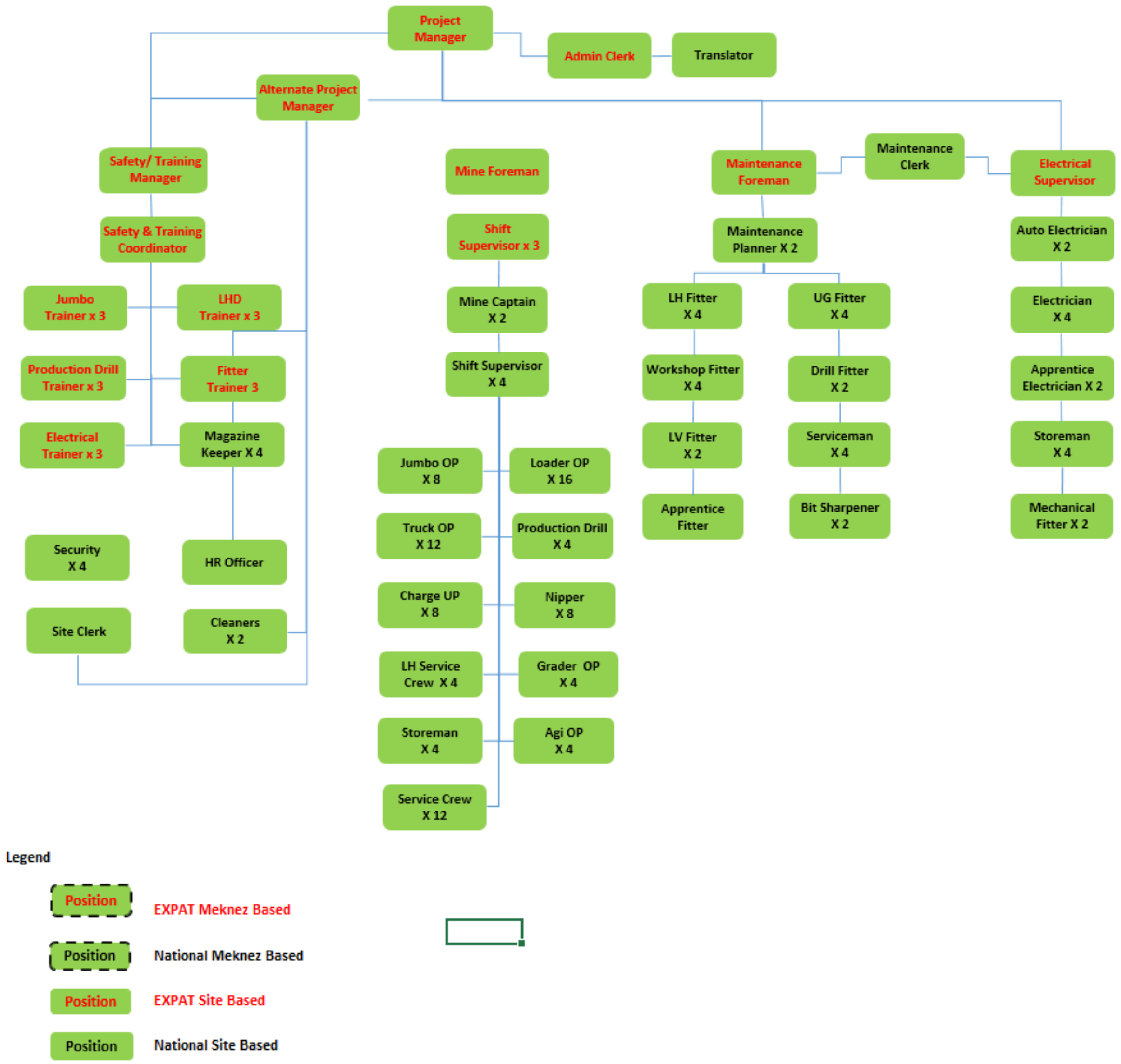


Figure 8-5 Mining contractor organisational structure

8.1.6 Remuneration and Working Hours

Atlas Tin engaged the services of a mining industry HR and recruitment consultant to assist in the understanding and development of recruitment strategies applicable working rosters and rates of remuneration.

8.1.7 Remuneration

Remuneration in the mining sector in Morocco is subject to the laws in force, particularly the labour code and the status of a miner. Remuneration is composed of a salary and a complementary salary which varies depending on the socio-professional category.

Employees that work in underground mines receive an additional 7% premium to the base salary.

An employee's base salary increases with the length of service of the employee: at 2, 5, 12, 20, and 25 years of continuous service with a company results in salary in salary increasing by 5%, 10%, 15%, 20%, and 25%, respectively. These increases are set by the minimum pay raises outlined in the labour code. Employees that are not housed in the mine camp are paid a housing allowance.

The mining activity brings out three main socio-professional categories that are: Managers, ETS (Employees, Technicians, Supervisors), and workers.

The Category of Managers

Managers constitutes the senior management. These are typically engineers and the executives of the company. In general, they have a degree (Bac +5) or a university (Master, PhD). Experience and level of further education are used to negotiate salary. Table 8-4 summarizes salaries of managers as they relate to the project.

Table 8-4 Category 1 managers (US$)

Category 1	Academic Level Required	Position	Base Salary (13 months per year) Dh	Performance Bonus Dh	Housing Allowance Dh	Transport & Remote Allowance Dh	Representation Allowance Dh
Superior Manager	Bac+5, or PhD, or +experience	Project coordinator, Mining manager, Project Director, Mining Director	more than 30,000 ($3,300)	up to 30%	1,500 ($165)	1,500-6,500 ($165-$715)	5,000-10,000 ($550 – $1,100)
Senior Manager	Bac+5, or PhD, or +experience	Mining engineer Processing Engineer Maint Engineer Geology Manager Admin Manager	20,000-30,000 ($2,200 – $3,300)	up to 30%	1,500 ($165)	800-4,500 ($90-$500)	4,000 ($440)
Principal Manager	Bac+5, or PhD, or +experience	Production Engineer Mechanical Engineer Electric Engineer	16,000-20,000 ($1,760-$2,200)	up to 30%	1,500 ($165)	800-3,500 ($90-$380)	2,000 ($220)
Confirmed Manager	Bac+5, or PhD, or +experience	Production Engineer Geotech Engineer Mine / Exploration Geologist	13,000-16,000 ($1,430-$1,760)	up to 30%	1,500 ($165)	800-3,000 ($90-$330)	1,000 ($110)
Junior Manager	Bac+5 or Masters or PhD	Beginner	10,000-13,000 ($1,100-$1,430)	up to 30%	1,500 ($165)	800-2,500 ($90-$275)	-

Typically, managers receive a bonus either annually or biannually. Transportation or remoteness allowance is paid in the monthly salary and its value depends on the remoteness of the mining site:

- If housing is provided in the mine site remoteness allowance is applied but not transportation and housing allowance.
- If housing is not provided transportation is on cost of the company, housing allowance is applied, and remoteness allowance isn't applied.
- The amount of representation allowance is based on the level responsibility in the company.

ETS (Employees, Technicians, Supervisors)

This category constitutes the middle management and technicians. As with managers, remuneration depends on the degree and level of individual experience. The following table (Table 8-5) outlines remuneration levels for ETS employees relevant to the project.

Table 8-5 Employees, technicians and supervisors (US$)

Category 1	Academic Level Required	Position	Base Salary (13 months per year) Dh	Performance Bonus Dh	Housing Allowance Dh	Travel & Remote Allowance Dh	Responsibility Allowance Dh
Expert Supervisor 2	Bac +3/4 + experience	Lab supervisor Store supervisor Admin personnel Security Supervisor, Senior Chef	8,500-12,000 ($930-$1,320)	0-30%	1,000 ($110)	500-2,400 ($60-$260)	1,100-2,000 ($120-$240)
Superior Supervisor	Bac +3/4 + experience	Mining supervisor, Workshop Supervisor	7,500-9,000 ($820-$1,000)	0-30%	1,000 ($110)	500-2,000 ($60 – $220)	800 ($90)
Confirmed Supervisor	Bac +3/4 + experience	Sector supervisor	6,000-7,500 ($660-$820)	0-30%	1,000 ($110)	500-1,800 ($60-$200)	500 ($60)
Basic Supervisor	Bac +3/4	Beginner supervisor	5,000-6,000 ($550-$660)	0-30%	1,000 ($110)	500-1,750 ($60-$190)	
Technician/ Employee	Bac+2/3	Mine technician, Maintenance Crew Admin Crew Lab Crew etc	4,000-5,000 ($440-$550)	0-30%	1,000 ($110)	500-1,750 ($60-$190)	400 ($40)

The term "employee" has a specific meaning in Morocco referring to those that have formal but low level studies (baccalaureate) and some months to two years degree of administration proficiency; this is more about basic secretary and office duties. They typically perform activities (technical secretary), cashiers, recording activity parameters in software, and basic admin day to day duties.

Technicians typically have a degree from schools and institutions of specialized training of two years of baccalaureate. This group includes: mining technicians, mechanics, electricians, geological samplers, lab crew, processing crew, maintenance, and basic day to day mining activities.

Supervisors have a degree from a school or university with two to four years of studies in specialised areas such as mining, geology, maintenance, processing, environment, accounting, IT, human resources, etc.

ETSs receive a monthly performance premium that is from 0% to 15% of the base salary. They also receive an annual bonus that varies between 0% and 20% of base salary.

Transportation and remoteness allowances are applied in the monthly salary as follows:

- If housing is provided in the mine site remoteness allowance is applied but not transportation and housing allowance.
- If housing is not provided, transportation is on cost of the company, housing allowance is applied, and remoteness allowance isn't applied.

Workers

The minimum wage in Morocco is 13.46DH/hr, (US$ 1.50/hr) which relates to unskilled work. Personnel at this level are referred to as "workers". Table 8-6 summarizes the wages of workers in the Moroccan mining sector.

Table 8-6 Workers (US$)

Category 1	Academic Level Required	Position	Base Salary	Performance Bonus	Housing Allowance
Qualified worker	Certificate of professional qualification or experience	Miners, process plant workers, workshop, drivers, store crew, cashier	3,500-6,000 ($	10%	600 ($70)
Specialised worker	Basic Degree & Experience	LV, HV Drivers fuel pumps, Welder, Sampler, Mechanical, Electrician	3,000-4,000 ($330-$440)	10%	600 ($70)
Man oeuvre worker	Elementary school level	Trainee, assistant, guard	Minimum wage	10%	600 ($70)

There can be three levels of workers, and productivity premium of between 75 and 15% can be awarded. Exceptional premiums can also be awarded to workers for an occasional exceptional work.

If housing is provided in the mine site there is no housing allowance nor are family allocations; if housing is not provided housing allowance and family allocations are paid to the worker.

Other examples of advantages that might be applied are:

- travel allowance once during annual leave
- dirty work allowance, soap or equivalent
- allowance of special occasions; sheep, miner's feast, toys for kids, once a year
- heating allowance between November and February if housing isn't heated.

Experience and continuous training/studying can allow an employee to gain levels in the organization and upgrade to another category. The company benefits from new competencies with an improvement in remuneration.

8.1.8 Organization of working hours

Since 2004, the statutory working hours have been set at 44 effective hours per week or 2,288 hours per year with a weekly break. There is an obligation to have a rest period of at least 20 minutes after a maximum of 5 hours of continuous work. It is not considered working time.

Travel time between the place of accommodation and the place of work is not considered as working time. Extra working hours are regulated and require prior authorization by company management.

For employees that don't work shifts, 44 working hours per week is set as a basis. 44 hours spread over 5 days or 6 days with a weekly break of 1 to 2 days.

If there is requirement to work extra hours, they are converted to time off in lieu (TOIL) for managers and ETS's. However, workers have a choice between converting them into TOIL hours or being paid for them as per the labour code.

Atlas Tin has developed a remuneration model based on three, eight-hour shifts and adopted salary ranges appropriate to the nature of the roles required, education, training, experience and project location in accordance with Moroccan labour laws and industry rates.

8.2 Transport and Logistics

The Project site is located 51 km by road form the city of Meknes and a further 150 km to Rabat, the capital of Morocco (Figure 8-6). The location and transport infrastructure provides permanent all weather access to the project site and access to the major highway networks to the major port facilities in Casablanca.

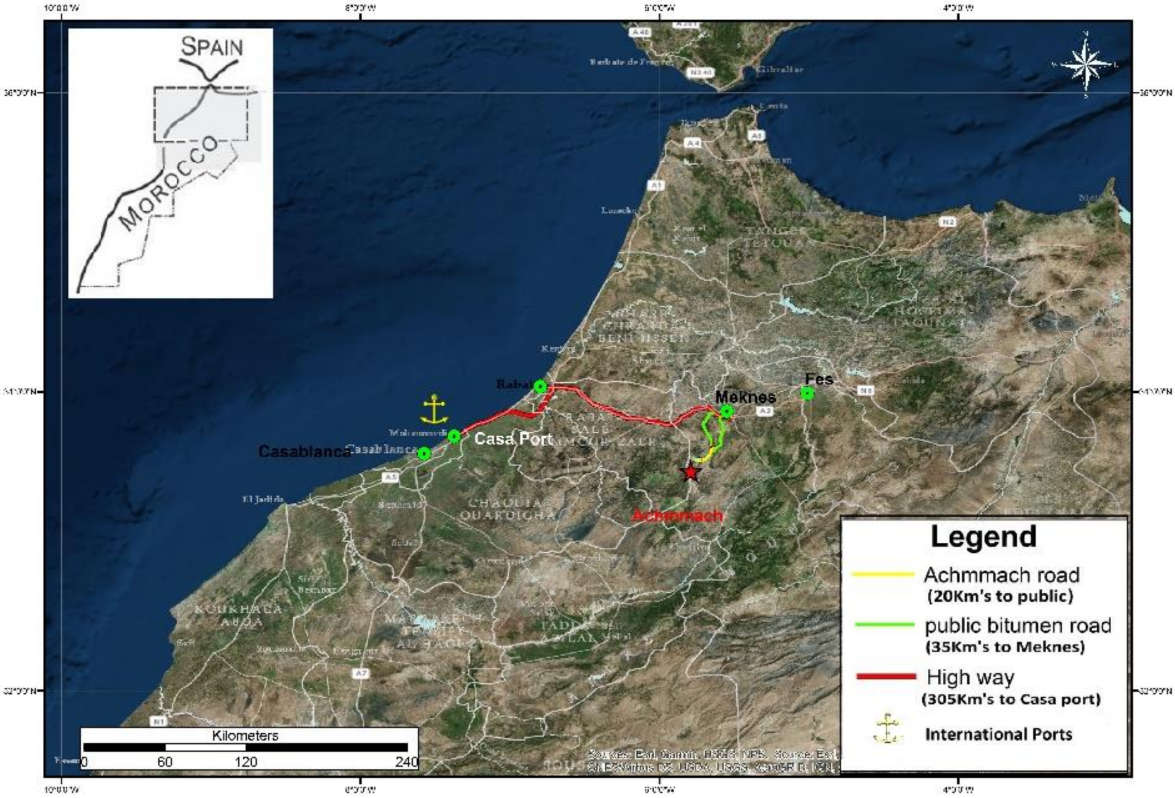


Figure 8-6 Achmmach location and road access.

Access to the site from Meknes is via by a section of sealed public road (31 km) and a further unsealed section of public road (20 km). The unsealed section of road provides all weather access to the Project site. A typical stretch of the unsealed road section is presented below (Figure 8-7).

Figure 8-7 Unsealed Section of the Achmmach access road.

The unsealed project access road is used by the local community. Atlas Tin in conjunction with the local communes has and continues to contribute to the ongoing maintenance of the road.

8.2.1 Achmmach Logistics and Transport

The ESIA (environmental and social impact assessment) undertaken by Kasbah Resources on behalf of Atlas Tin identified community concerns relating to road access and road safety during its public inquiry and engagement program. The community was positive about the road access being maintained by Atlas Tin and thus providing better access to the region.

An initial review of the traffic that will be generated by the project is shown in Table 8-7 below. to understand the potential impacts on the community and ensure adequate management of any impact or risks to the community are identified and addressed.

Table 8-7 Achmmach project traffic survey

	Total	LV	Bus	Truck
Management	61	61	-	-
Crew transport	397	305	92	-
Camp	16	2	-	14
Concentrate	61	-	-	61
Stores and inventory	27	4	-	23
Contractors	106	36	-	70
Miscellaneous	61	40	-	21
Total monthly trips to site	**729**	**448**	**92**	**189**
Average daily trips to site	24	15	3	6

The traffic survey identifies that there will be 24 round trips to site on a daily basis to support the transportation of crews, logistics and supplies to the operation. There has been no detailed rationalisation and combination of loads included in this survey between contractors and site supplies, other than crew transportation. It is expected that supply and logistics companies will use Meknes as the logistics hub where site supplies will be combined. Atlas Tin management will develop policies and practices for the site that minimise road transportation activity.

8.2.2 Road Diversions and Upgrade

Atlas Tin has continued to use the unsealed public access road since the establishment of the project site. During this time Atlas Tin has supported the ongoing maintenance of the road. Since the establishment of the project site Atlas Tin has trucked camp and infrastructure into the project site, implemented a large exploration drilling and resource development campaign and undertaken earthworks programs to establish site infrastructure. No road access issues have been identified to date.

Atlas Tin will look to improve the road access and road safety during mine construction and production. It is proposed that the 20 km long section of unsealed public access road to the site will require two small diversions the first being the turnoff point to the site to provide safe entry and exit turning for larger trucks (Figure 8-8).

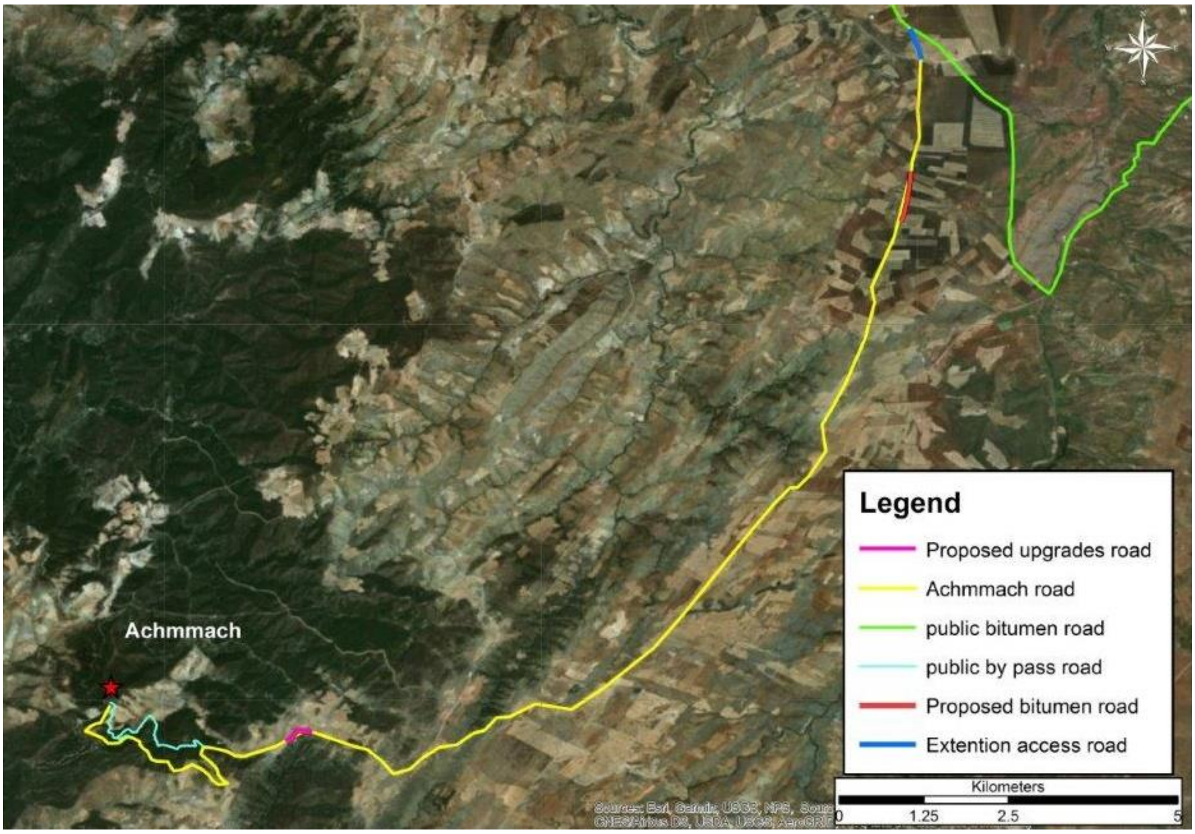


Figure 8-8 Local road access and upgrade

The second section will require a 200 m length of road to be moved away from a small village. In addition, it is proposed that this stretch of road will be bituminised to reduce road traffic noise and dust (Figure 8-8).

In addition to these two areas there are a number of small sections of the road that will require upgrading and widening to safely meet the projects traffic requirements. Atlas Tin will undertake these on an on-going basis during construction and mining operations. The mine will produce crushed waste. This material is proposed to be used for site infrastructure, mine back fill, road upgrade and ongoing road maintenance.

A further road diversion (Figure 8-8) is proposed for the project to take the public traffic away from the mine operating area.

8.2.3 Traffic Management

As part of its operational management plans Atlas Tin will develop a traffic management plan to ensure that Atlas Tin's activities are respectful of the community and address key safety concerns identified by the local communities. Such community agreements are common place in industry and will not impact supply to the project.

Transport Hours

It is common in industry for trucking and site deliveries in small communities to be scheduled outside of school start and finish times and during daylight hours. In addition, the site will schedule traffic around community events such as local market days and festivals. Such events will be identified by the community liaison officer and will be planned with the local communities.

Logistics Management

The combining / pooling of site supply requirements is a key driver in reducing traffic volumes. Atlas's supply and logistics team will work with logistics groups to maximise the efficiencies and reduce road traffic. Other than crew transport no freight forwarding efficiencies have been included in the traffic survey numbers. It is planned to have a delivery depot in Meknes from where site supply can be coordinated to ensure both the timing of deliveries are acceptable and so that loads can be combined where possible to reduce traffic.

Traffic Wardens

The Achmmach access road passes by the local community school (Figure 8-9). In addition to restricting site logistics deliveries during the start and finish of the school day, Atlas Tin will maintain a traffic warden on the road located near the school. A local resident will be employed in this role to ensure traffic passes through the area at reduced speeds and that the road is safe to pass. This is seen as a key community interaction point.

Figure 8-9 Local community school

8.2.4 Road Upgrade Costs

Atlas tin has reviewed a number of road options for the project and continues to explore opportunities to reduce its road footprint. The site access road already provides excellent all weather access to Achmmach and has been used by Achmmach during the exploration drilling phase of the project to deliver earthmoving equipment, diamond drill rigs and site infrastructure.

With the introduction of ore sorting Achmmach will generate excess crushed waste material suitable to be used in the construction of roads. This will be used to continually upgrade the road over the project life.

Atlas has allowed a total of US$ 1.2 million to address the key areas of the access road. This estimate is based on supplier indicative pricings. Construction and upgrading of roads and access ways have been included in the processing plant earthworks.

8.3 Information Management

8.3.1 Information Systems

The following sections describe the intended information management plan to address the requirements of the project team during execution and Moroccan based site and administrative office personnel during operations. Key considerations of the information management plan are the number and location of the user base, security, functionality and availability.

8.3.2 IT Infrastructure

The IT infrastructure system will be comprehensive in nature to enable reliable connectivity between the Achmmach site, the Meknes administrative office and beyond.

Local Area Network Design

The proposed IT infrastructure system site will cater for the peak workforce of approximately 350 personnel, including approximately 20 Atlas Tin employees located in Meknes and on-site contractors. The onsite network will comprise distinct network segments to accommodate the varying security requirements of personnel accessing the system and will require the implementation of two separate local area network systems.

The primary business network will comprise a traditional hard wired ethernet system which caters for high speed access between desktop PCs, laptops, printers and servers etc. This network will also have WiFi access capability to allow authorised staff to access the network resources remotely using WiFi. The second network segment will be a WiFi only system which only allows access to the public internet. No access to the secure servers or other resources will be available from this system. Access to the public internet will be subject to bandwidth throttling and daily quotas to ensure equitable use of the system. Access to certain content may be blocked or limited for security and bandwidth conservation.

The Meknes office will not require the split security capabilities required at the mine site. It has good access to the public internet, and the IT infrastructure will therefore be designed as a single segment local area network (LAN).

The structure of the network segments is illustrated in Figure 8-10.

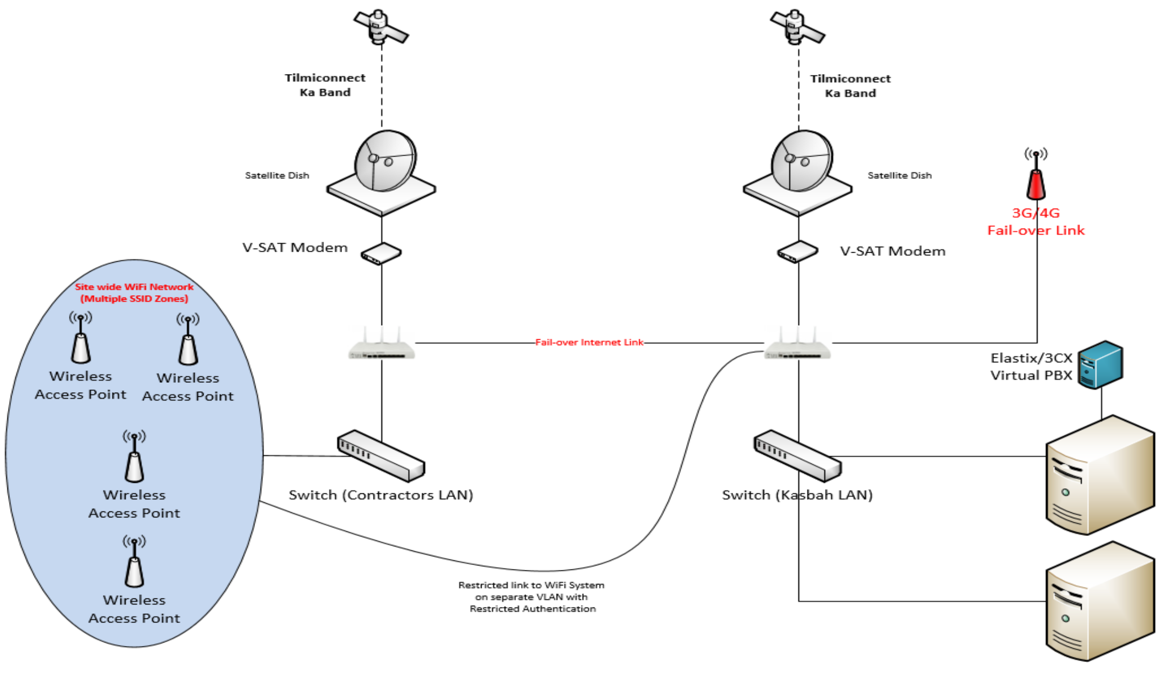


Figure 8-10 Site network segment structure

Internet Connectivity

As the Achmmach site is not within range of traditional internet networks, connectivity to the public internet will be achieved using two Ka band satellite links provided by a domestic telco provider. The satellite connectivity remains the fastest and most reliable way of linking the site to external locations. Due to the location of the site other technologies such as Wireless 3G/4G are either unavailable, too restrictive or cost prohibitive. Recognising this situation may be changed over time, the Information

Management plan has been developed to be readily adaptable to new connectivity options as they become available.

The Meknes office is currently serviced by ADSL2+ and Fibre Optic links which are serviced by MarocTel, the main telco provider in Morocco. The same Firewall/Router/Virtual Private Network (VPN) devices as used at site will be used to ensure secure VPN links and allow seamless authentication with Microsoft Windows Active Directory and Microsoft Azure.

Server Systems

The server systems will be comprised of a dual server system utilizing fail-over clustering and Hyper-V replication to ensure maximum availability, sufficient storage and processing capacity.

The servers will have fault tolerant and redundant disk storage systems, power supplies and internal fan systems. The fail-over cluster provides continuous, real-time replication of data and virtual servers between the two hosts to maximise system up-time and availability.

A single server with identical specification to the two units at site will be installed in Meknes so that it can be deployed as an emergency disaster recovery measure at site if an emergency event occurred.

Backup systems

The backup systems will comprise a multi-tiered approach utilizing both on-site and off-site backups. In addition to the fail-over capabilities of the servers described above, a comprehensive and multi-pronged backup solution be deployed. This system will comprise the following elements;

- on-site backup to alternative storage medium (High capacity NAS Storage)
- on-site backup to portable backup media (RDX backup cartridges or HP Ultrium cartridges)
- replication of key data to off-site servers over the satellite internet links.

In addition to real-time replication of data between site and Meknes, the RDX or Ultrium devices can be used interchangeable to backup and/or restore data from one site to the other.

Telephony Solution

The proposed telephony solution makes use of state-of-the art SIP based Voice over IP PBX software to deliver a fully featured system that integrates on-premise handsets with mobile phones as well as off-site links and IT system integration.

Cost Estimate

A costed estimate has been included in the financial model for IT infrastructure, including:

- 3 servers (and related software)
- 65 workstations
- 15 laptops
- telephony
- peripheral equipment
- installation and configuration.

Security and Local Technical Support

System backup tapes (RDX Cartridges) will be taken offsite at regular intervals in a rotating schedule. There will be a dedicated secure communications room with air conditioning and a reliable power supply on site.

Allowance has been made for an appropriate number of licences of Threat Track Vipre Endpoint Security Antivirus Software and GFI Webmonitor which provides App Whitelisting and application management. Other endpoint application management is also handled by Microsoft Windows Group Policies.

Technical support will be managed and coordinated by an onsite IT Assistant and Administrator assisted and supplemented as required by a local provider, yet to be determined.

8.3.3 IT Software

In addition to IT infrastructure, allowances have also been included for the following software supporting operations.

Microsoft Office 365

Allowance has been included for subscriptions for 80 users totalling US$18,000 per annum.

Enterprise Resource Planning Software (SAP B1 or Pronto Xi)

Allowance has been included for implementation costs of US$180,000 and ongoing annual licensing fee of US$11,000. In addition, an allowance of US$30,000 has been included for the development of macro-based management and financial reporting templates.

Surveying, geological modelling and mine planning software (Geovia Surpac and Deswik)

Allowance has been included for 2 GEOVIA Surpac licenses with one-off costs totalling US$162,400 and annual license fees of US$23,500. In addition, allowance also included for 4 Deswik licences with one-off costs totalling US$152,000 and annual licence fees of US$20,300.

9 HEALTH, SAFETY, ENVIRONMENT AND COMMUNITY

9.1 Introduction

Atlas Tin has the exploration and mining rights LE 332912, covering an area of 11 square kilometres which is referred to as the 'concession' in this report.

The concession is located in the rural districts of Ras Ijerri, Jahjouh and Ait Ouikhalfen, situated in the El Hajeb Province, part of the Meknès – Tafilalet region, and which is situated in the northeast part of the central plateau of the Atlas Mountains. The nearest city is Meknès (55 km to the northeast) and the nearest town is Agourai (20 km to the east).

The Achmmach tin deposit was discovered by the Moroccan National Office for Mineral Exploration (BRPM) in 1985 and from 1991 onwards BRPM undertook extensive exploration and evaluation work. In 2006, Kasbah entered into a joint venture arrangement with the National Office for Hydrocarbons and Mining (ONHYM) to further explore the deposit.

The general environmental context of the region is a sparsely populated mountainous area with valleys and plateau features. The area is characterized by forest areas of pine and oak and open land which is used for growing cereal crops and seasonal grazing of animals. There are approximately 300 people living in the concession area with five households located in the immediate vicinity of the future mining facilities. These five households will be resettled prior to construction.

9.2 Environmental and Social Impact Assessment Process

Artelia prepared the ESIA (environmental and social impact assessment) in compliance with Morocco's environmental regulations, and in particular law 12-03 relative to environmental impact assessment and Decree n° 2-04-564 regarding public involvement.

The ESIA scoping report was prepared in 2011 and presented to the National Committee for Environmental Impact Assessments (CNEIE) on 15 June 2011. The committee accepted the report and issued terms of reference for the environmental impact assessment.

The ESIA assessment was carried out during the period May 2011 to June 2013. This included completion of a two-season environmental baseline survey, the first being in May 2011 and the second in October 2011. A draft final ESIA was prepared as part of the PFS (pre-feasibility study) and submitted to the CNEIE in September 2012. A second version of the ESIA was prepared as part of the DFS (definitive feasibility study) that was undertaken in 2014. This included the findings of the social baseline survey that was carried out in April 2013. The final version of the ESIA integrates the answers to the CNEIE comments raised after its subsequent review in October 2013 and was approved by the CNEIE at its meeting of 11 March 2014.

9.3 IFC Performance Standards on Environmental and Social Sustainability

The ESIA was also been prepared to comply with the International Finance Corporation (IFC) performance standards on environmental and social sustainability (1 January 2012). These are similar to the ten Equator Principles.

The project is a greenfield mining project and therefore is considered as a Category A project.

The rationale for the triggering of, and ensuring compliance with, the performance standards are summarized as follows:

Performance Standard 1: Social and Environmental Assessment and Management System

The project facilities cover an area of 73.8 ha and which is largely modified habitat comprising mainly deforested land. The land is currently used for seasonal grazing of animals by local people living near the site and a small area (2.3 ha) of Aleppo Pine woodland (which is not the natural habitat but has been artificially introduced) and 1.3 ha of cork oak forest. There are approximately 300 people living in the concession, which includes approximately 20 people in five households who will need to be relocated. An ESIA including an environmental and social management and monitoring plan (ESMMP) has been prepared (Appendix 9A).

Performance Standard 2: Labour and Working Conditions

The Project will require the recruitment of a local workforce for both construction and operation of about 350 people.

The ESMMP includes requirements for the preparation of a recruitment policy and plan which include the following themes:

- working conditions and management of worker relationships
- protecting the work force
- occupational health and safety
- workers engaged by third parties
- supply chain.

Performance Standard 3: Resource Efficiency and Pollution Prevention

The project is in proximity to surface and groundwater resources used by local people. The project will also require the use of these water resources. There is a risk that the water resources may be depleted and/or polluted by the project activities. To address this risk, the facilities have been designed to minimise water use, maximise recycling of water and the optimise supply water to the project by ensuring that the facilities are designed in collaboration with and approved by the water basin agency (Agence du bassin hydraulique du Sebou – ABHS). The ESMMP includes plans to manage water use and pollution prevention.

Performance Standard 4: Community Health, Safety and Security

The ESIA includes social baseline information on community health safety and security. The impact assessment evaluates the impact of the project on community health, safety and security and establishes preventive and control measures. The ESMMP includes plans for the implementation of the preventive and control measures.

Performance Standard 5: Land Acquisition and Involuntary Resettlement

The project does not involve land acquisition. The current land tenure of the areas where mining facilities will be located is collective (common) land for the cleared areas and forestry land for the wooded areas. Atlas Tin will pay rent to the forestry commission for the use of the forestry land and will also pay rent on the use of collective land. There will also be a need to relocate five households (containing seven families) located close to the future process facilities. For the resettlement and compensation process, Atlas Tin will initiate discussions with the people concerned. If an agreement is reached this would be ratified by the Ministry of the Interior (MoI). If an agreement cannot be reached the issue will be handled by the MoI, which will determine the level of compensation payable by Atlas Tin, with the assistance of the provincial administration in El Hajeb. The ESMMP includes a resettlement framework which outlines the general principles to be followed by Atlas Tin to ensure that resettlement will be carried out in line with the PS5.

Performance Standard 6: Biodiversity Conservation and Sustainable Natural Resource Management

The concession encompasses areas of natural habitat (Cork Oak woodland) and modified habitat (Aleppo Pine woodland and cleared areas). There are no areas of protected habitat or critical habitat in the project area of influence. The project has been designed to minimise impact on woodland and project facilities footprint encompasses mainly the cleared land. The ESIA evaluated the impact on biodiversity.

Performance Standard 7: Indigenous Peoples

The ESIA included interviews with local authorities and a social baseline survey. It has been established that there are no indigenous peoples in the project area of influence. The people in the region are referred collectively to as the Guerrouane du Sud but this group of people is not to be confused with indigenous people (see description of natural and human environment in the following for more details).

Performance Standard 8: Cultural Heritage:

Interviews with local authorities and the social baseline survey have established that the only cultural heritage site near the project is the Sidi Addi peak. The site, although not an officially recognized cultural heritage site, has some importance for local people. Each year there is a gathering of local people at site on 8 April. The site, although located within the concession, is at sufficient distance from project site to be unaffected. The ESMMP includes measures to ensure that Sidi Addi will be protected and access by local people will not be hindered.

9.4 Project Components and Project Planning

The development of the Achmmach mine will comprise the following main components:

- development of underground mining facilities
- site preparation activities including upgrading of sections of the mine access road
- construction of a 60 kV power line to the project site
- construction of the ore processing facility, laboratory, administrative buildings, workshops and stores
- construction of a tailings storage facility
- construction of potable water, process water and storm water systems
- a mining operation producing a nominal 7, 000 tonnes of tin concentrate per year which will be transported from the mine by truck
- mine operations which will be undertaken by a suitably qualified mining contractor.

The layout of the facilities is illustrated in Figure 9-1.

In terms of work allocation and responsibilities, the overall project organisation during construction can be summarised as follows:

- Atlas Tin, as the project owner, will be responsible for the project implementation, general site management and coordination, operation of the processing plant and management of mine waste and discharges.
- A suitably qualified mining contractor will be engaged to perform the underground mining scope of works.
- Selected construction contractors will be required to carry out the site preparation works, road improvement works, construction of buildings and process facilities and any other civil engineering works.
- A mine manager will oversee establishment of mining operations.
- A specialised contractor for the storage and use of explosives will be engaged.

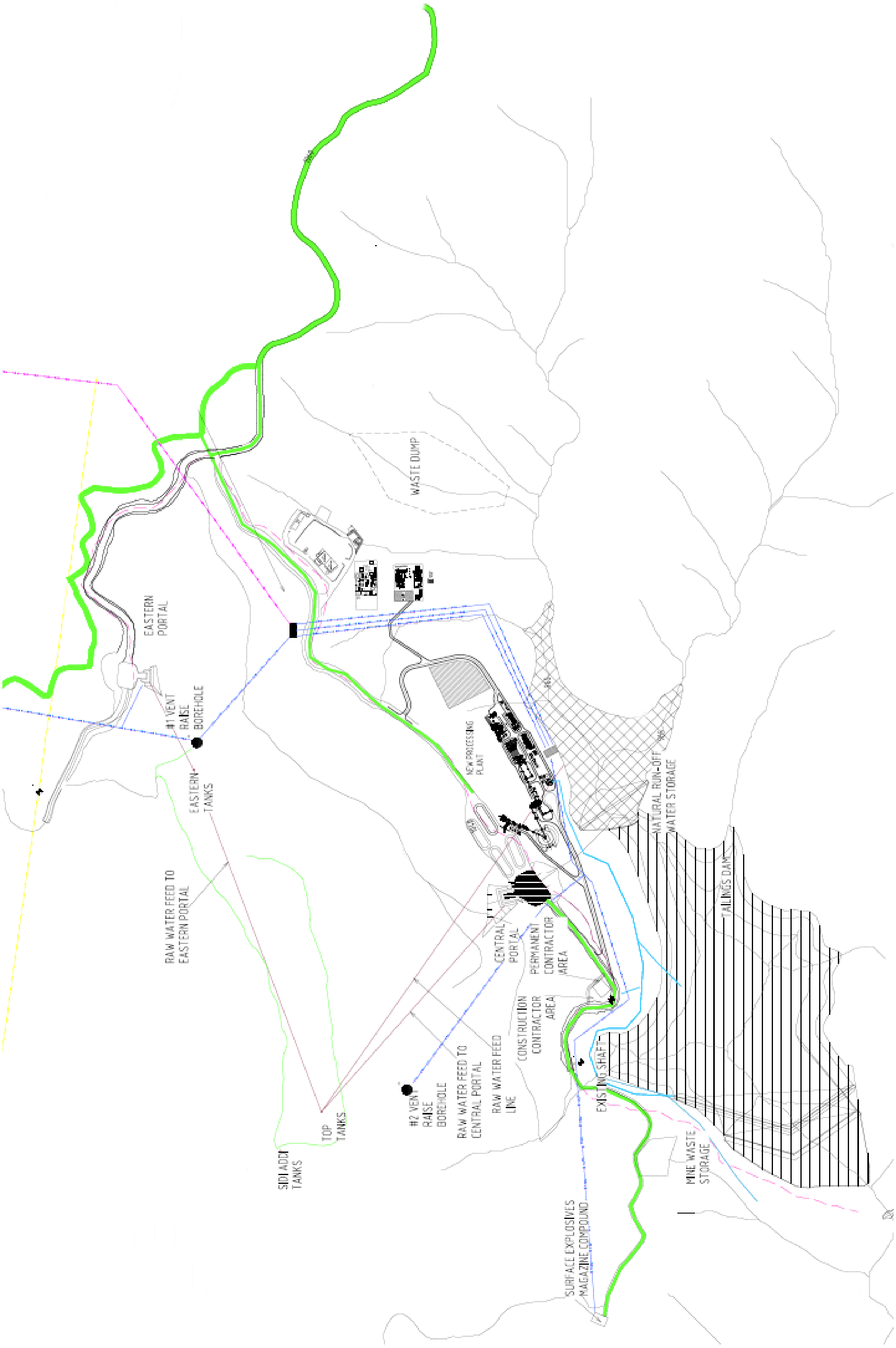


Figure 9-1 Preliminary site layout plan

9.5 Description of The Natural and Human Environment

Baseline studies have been undertaken to support the Environmental and Social Impact Assessment (ESIA) required to obtain approvals associated with the development of the Achmmach project. The ESIA was prepared by Artelia Eau & Environnement in association with Artelia Maroc and with the assistance of Moroccan specialists for the environmental baseline survey.

The physical, social, economic and cultural baseline has been characterized for the project using primary information gathered in the field, and secondary information from official sources such as government records. Field studies and data gathering for the baseline studies were undertaken between 2011 and 2013. Some baseline data was obtained prior to 2011 by other international specialist consultants.

9.5.1 Abiotic Baseline

Climate

The project has a semi-continental Mediterranean climate. Winters are cool and wet and the summers hot and dry. Temperatures range from -2ºC to 43ºC. The project is located at an altitude of 1,085 m above sea level, with an estimated average annual rainfall of approximately 1,000 mm, with between 80 and 100 days of rain per year.

Topography

The project is in a mountainous area. The altitude of the Achmmach forest ranges from 550 m to 1,230 m. The main topographic feature within the concession area is a large elongated hill which takes up most of the concession area. The crest of the hill is orientated from east to west and the highest point, which is known as Sidi Addi, is at an altitude of 1,230 m above sea level.

Seismic Activity

The North of Morocco and the Atlas Mountains are known areas of seismic activity. The project is located in an area where historically seismic activity has reached a maximum intensity of VI on the MSK 1964 scale. The facilities have been designed taking into account this constraint.

Geology

The general underlying geology of the area is characterized by a series of schists, siltstone, sandstone and breccias that extend over the Achmmach district. Three northeast orientated regional scale faults transect the Achmmach concession and these are related to the general mineralization in the project area. These are the El Hammam fault, the Achmmach fault and a third unnamed fault which is possibly an extension of, or is related to, the Achmmach-Bou el Jai fault.

Land Use

The land-use in the project area comprises forested and deforested areas. The deforested areas are used for agriculture. Aleppo pine woodland (an introduced species) dominates the land use in the concession equating to 33% of the land while the scrub / pasture land and rain fed cereal crops equate to 25% and 24% respectively.

There are no protected areas near the concession. The nearest environmentally protected area is the Ifrane Park, which is approximately 40 km southeast of the project area and beyond the project's area of influence. The park is upstream of the project area.

Hydrology

The project is located in the Sebou River catchment basin, which represents 30% of the total surface water in Morocco. This catchment is comprised of six sub-catchments. The project is located in the River Beht sub-catchment which drains 363 million cubic metres of water per year, representing 7% of the total water drained by the Sebou catchment. The Beht river catchment has a total surface area of 9,000 km² and the confluence with the Sebou is on the Gharb plain. The River Beht is located 4 km from the western boundary of the concession. Surface water from within the concession area flows towards twelve local streams which extend outside the project concession.

Hydrogeology

The groundwater in the Sebou river basin represents 20% of Morocco's groundwater reserves. The project is located near the southwest perimeter of the basin in an area where the geology is characterized by the presence of schist and a lack of recognised major aquifer. The nearest aquifer of importance is the Agourai aquifer which is located about 20 km to the northeast of the concession.

Groundwater resources within the concession appear to be limited to small isolated shallow aquifers fed by the seasonal infiltration of rainwater and geological structures incorporating dyke features that act as water traps. It is thought that that the different zones containing underground water are isolated with minimal interconnections. A limited amount of deep groundwater has been recorded in the concession near the Sidi Addi and was encountered during underground exploration in 1997. Monitoring in 2007 to 2009 saw a decline in water level, which coincided with the commencement of underground dewatering in the neighbouring fluorite mine. A survey of the concession located 73 shallow wells (0.5-3m) in close proximity to the project area.

Water quality in the concession is classified as fresh water with a pH range between 6.5 – 8.5 and a conductive range between 114 – 716 µs/cm. Some naturally elevated levels of Bismuth where noted in the groundwater. Local water supplies are used for potable water and agricultural purposes.

9.5.2 Biotic Baseline

Flora

There are five types of habitat on the concession area; these are Aleppo pine woodland, cork oak woodland, cleared areas, rocky outcrops and riverine habitats. The Aleppo pine woodland covers 75% of the concession area. This however is not the natural vegetation but has been artificially introduced to replace the natural cork woodland that has been progressively cleared by local people.

The project's flora inventory has identified 237 plant taxa in the concession area comprising 24 endemic and 8 rare taxa. The rare taxa are present in low numbers and sporadically distributed on the wooded and rocky outcrop habitats which the project will not disturb.

Terrestrial Fauna

The fauna assessment of the project area identified reptiles (19 taxa), amphibians (6 taxa), mammals (17 taxa) and birds (98 taxa) present in the concession area. In terms of rare or endangered animals, tracks of the Golden Jackal (classed as vulnerable by the IUCN) were observed and a number of important, rare and remarkable birds are thought to be present. These include two endemic species the Moussier's Redstart and Levaillant's Woodpecker; and six rare species-Montagu's Harrier, the Hen Harrier, the European Garden Warbler, the European Siskin, the Common Crossbill and the Ortolan

Bunting. The project is not expected to have a direct impact on these species as very small areas of woodland will be impacted and only a small part of the concession area will be developed.

Aquatic Invertebrates

The aquatic invertebrate population is dominated by Diptera (flies and mosquitoes) which comprise over 30 taxa including 13 threatened species. The second most abundant group is the Trichoptera comprising 24 species. The rest of the population comprises Ordonata, Molluscs, heteroptera and crustaceans.

9.5.3 Social Baseline

The project area is located in the province of El Hajeb in the administrative region of Meknès-Tafilalet that comprises five provinces: Meknès, Ifrane, Khenifra, Errachidia and El Hajeb (Figure 9-2). The project area concerns three districts, namely: Ait Ouikhalfen, Jahjouh and Ras Ijerri, which are located inside the Agourai circle, in the province of El Hajeb. The nearest city is Meknès (40 km to the northeast) and the nearest town is Agourai (25 km to the east).

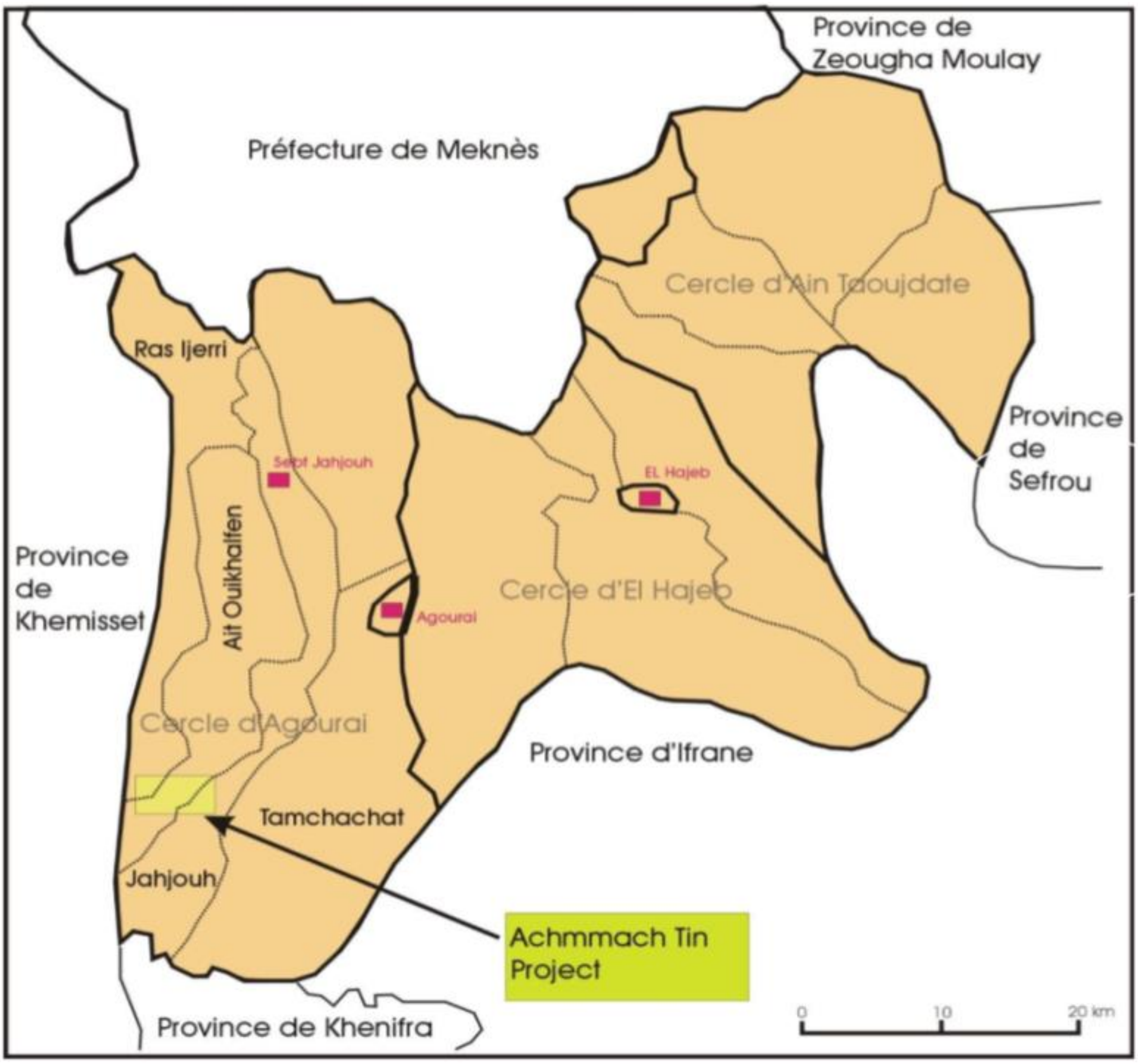


Figure 9-2 Administrative Zones – Province of El Hajeb

The total population of the three rural districts associated with the project is estimated at 19,700 people. Ras Ijerri and Sebt Jahjouh are the two principal urban centres which concentrate the majority of the population. Within the concession area there are 50 households representing an estimated 300 people (Figure 9-3).

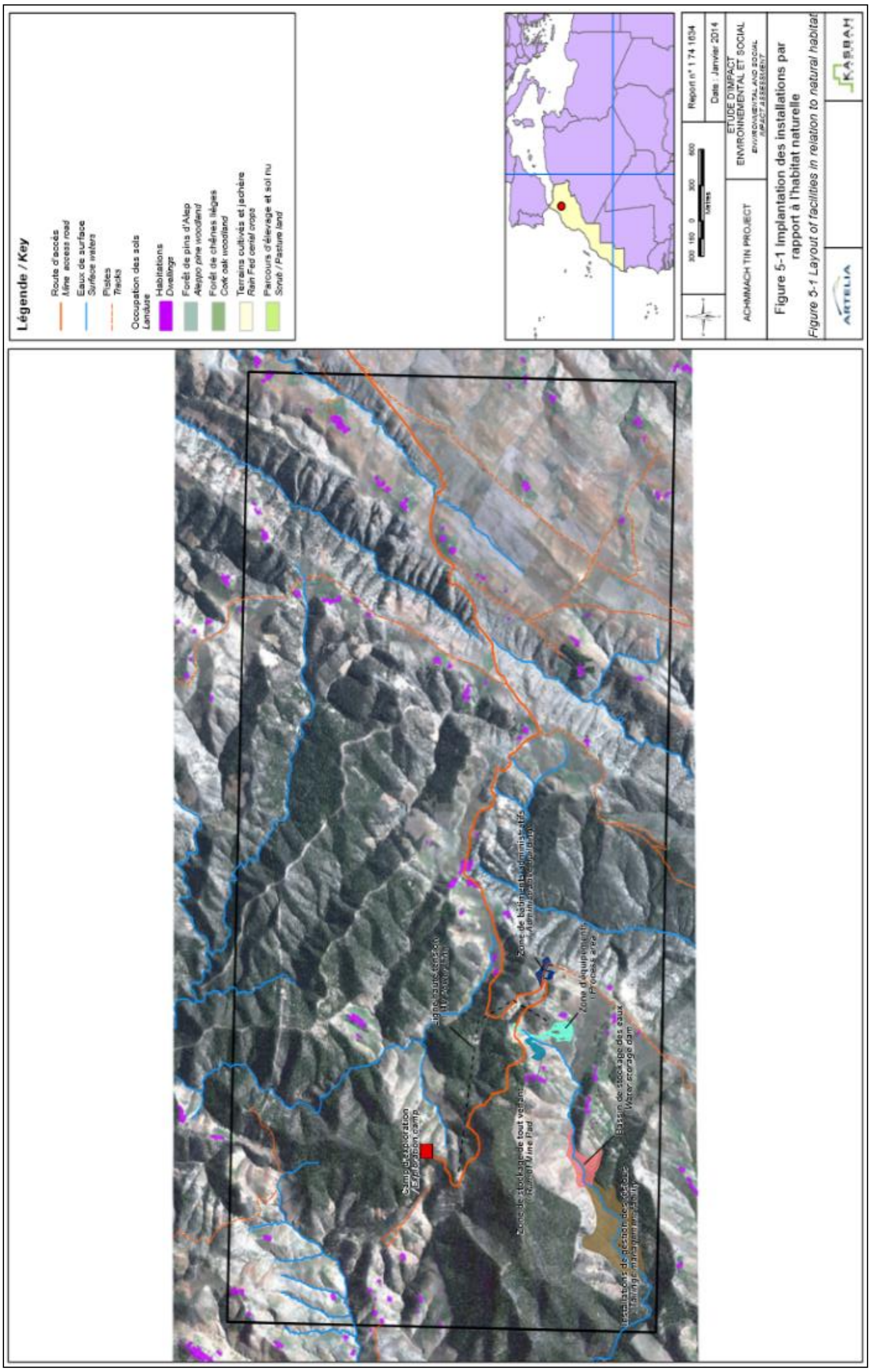


Figure 9-3 Layout of facilities in relation to dwellings and natural habitats

The project does not involve any direct land acquisition. The current land tenure associated with the project area is a combination of collective (common) land for the cleared areas and forestry land for the wooded areas. Rent will be paid for the use of both land categories.

Where practical the project has sought to minimize adverse socio-economic impacts and improve the livelihoods and standards of living of displaced persons. Five households (seven families) located close to the future process facilities will be relocated. Compensation for loss and resettlement will be paid in accordance with laws and current practices in Morocco, as well as the IFC Performance Standard 5 on land acquisition and involuntary resettlement.

Household consultations have shown a positive perception of the project among the local population. The project is seen as an opportunity for local development. The inhabitants and local authorities expect the project to be a source of job creation (especially for young people) and poverty reduction (through social infrastructures such as schools, health centres and access roads).

Main issues of local concern are:

- employment opportunities (in an area where the unemployment rate is very high, and the labour is not skilled)
- working conditions (workers' rights)
- pollution and water scarcity
- compensation rights for relocation
- health impacts (especially because of dust and water pollution)
- accident risks because of an increased road traffic.

It is expected that most unskilled positions at the mine will be offered to residents within the vicinity of the project, these being the communes of Jahjouh, Ras Ijerri and Ait Ouikhalfen.

Demographic Aspects

According to the last official census conducted in 2004, it was determined that El Hajeb province official populations was 216,388 and the three districts within the project area of influence, had a population of 17,761. Between 2004 and 2012 the population growth across the three districts was estimated at 12.16%. The Jahjouh urban area represents approximately 50% of total headcount.

The three districts have similar population age structures, with a majority (62% on average) of the population being of legal working age. The high proportion of working population is a challenge for the local economy and explains in part the high expectations regarding the project in terms of local employment.

The project is located inside a relatively sparsely populated area, the population being scattered over the area. Population densities are relatively low, with an average for the three districts of 51 inhabitants per km^2.

From the analysis of recent satellite images and the validation of results with the survey of households a total of approximately 50 dwellings were counted within the concession. The households consist of an average of six people, meaning that the total population affected can be estimated to be approximately 300 people.

Ethnicity

The region is populated mainly by the Southern Gerrouane, which is a Berber tribe from the south of Morocco whose languages are Tamazight and Amazighe. The Southern Guerrouane are present in six districts including the three districts associated with the project. The Guerrouane's ancestors are the Masmouda, from which the Almohade dynasty was born, and are part of the Ait Yafelman Berber confederation.

Today, the only generic name that seems to bring together the population around a common identity is "Southern Guerrouane". The Southern Guerrouane do not meet the criteria used to define indigenous peoples as described in the IFC's performance 7 standard.

Each douar (camp or village) is associated with a fraction of the Gerrouane group (for example, Ait Hamou Moussa in Ouikhalfen or Ait Krat in Jahjouh), "Ait" meaning "son of". However, today, the douars cannot be considered as ethnically homogenous groups. The project field survey showed that there are many mixed couples (Arab-Berber) in the projects area of influence. Amazighe and Arabic are two spoken languages, Amazighe being the main language for 68% of households interviewed.

Education

Access to education and literacy are essential issues in the project region. Statistics taken directly from households and the latest territorial assessments of the districts (2010) show that literacy and schooling rates are lower than the national average.

In the study zone, 65% of people older than 10 and who live in the households that were interviewed have not gone to school and the illiteracy rate reaches 90% (Table 20.1). People who have followed higher education are rare exceptions: amongst the people interviewed, only one went to university. The closest universities are located in Meknès, which is 40 km away from the study zone. The geographical dispersion of the population and the isolation of the douars, associated with the lack of financial resources and road infrastructure, make it difficult for most children to get to school.

Table 9-1 Level of Education of the Population Surveyed during ESIA in 2013

Level of Education	%	% of women	% of men
None	65.5	79	51
Primary school	28.4	20	38
Secondary school	5.6	1	10
Higher education	0.5	0	1

(Source: assessment based on data collected in the field by Artelia April 2013, from 197 interviews; 101 women, 96 men.)

Health Care

Health care facilities in the projects area of influence comprise district health care centres some of which are equipped with maternity modules. Indicators and health care staff/population ratios are much lower than those recommended by the World Health Organisation, i.e., a minimum of 25 medical professionals for 1,000 inhabitants.

The district health care centres provide standard medical consultations, including gynaecological and paediatric care, essential care, vaccinations, prevention and control of endemic diseases, health protection activities for mothers and children, and monitoring of health for school pupils. The health care centres also provide outpatient care: mobile teams travel to the most isolated zones to perform medical consultations at home. Lastly, on top of their medical role, the district health care centres implement sanitary programmes such as disinfection operations, bacteriological analyses of drinking water and inspection of food service establishments.

In the project's area of influence, none of the health care centres in the rural areas were equipped with an ambulance. Because of the low reception capacity of the local health care centres, a lot of people go to Agourai, or even Meknès to receive the care they require.

The most common conditions affecting the region are respiratory diseases (a lot of asthma), cardiac diseases, diabetes, hypertension and allergies.

9.5.4 Archaeological Baseline

There are no archaeological or formal cultural sites in the project area or in the vicinity. However, there are a few spiritual sites in the concession area and its vicinity, among which the Sidi Addi peak, which has a spiritual significance for local people as it is considered as the mausoleum of the Saint Addi. Many of the people consulted confirmed that they regularly visit this site during the year for prayers or tributes.

During community interviews, a number of other spiritual sites were noted, all outside the concession but close to the project. No impact is predicted on the Sidi Addi or the other sites.

9.6 Environmental Management

9.6.1 Environmental Design

The environmental design for the project area is based on Moroccan Law and practices, as well as International Finance Corporation (IFC) Performance Standards, IFC EHS Guidelines for Mining, Equator Principles (2006) and three key International Conventions (Bern Convention, Bonn Convention and Stockholm Convention).

9.6.2 Environmental Impacts and Mitigation Measures

Impact on Land Use – Just need final percentage

The establishment of the project will require land-use of an area covering 74 ha; (2.3% of the concession area). The facilities will be constructed on areas of land currently used mainly for the seasonal grazing of animals (70 ha), a small area (2.5 ha) of Aleppo woodland (modified habitat) and a small area (1.5 ha) of cork oak woodland (natural habit).

Impact on Water Resources

There are four risks with respect to impacts on water resources;

- reduced water availability for the local population due to groundwater extraction
- risk of contamination of water resources from the discharge of sanitary, domestic and process wastewater
- risk of contamination of groundwater by seepage from the TSF
- increased turbidity in the seasonal streams.

The means of supplying water to the mine will be established in collaboration with, and approved by, the water basin agency (ABHS). The annual mine water consumption is expected to be around 270,000 cubic metres per year, which translates as approximately 57 m^3/h. An assessment of water consumption indicates that the project will be in surplus on an annual basis. However, there will be a seasonal deficit during the summer months when it will be necessary to obtain water from external sources.

The project water supply will rely on surface run-off that is harvested during the winter season and stored in a dedicated water storage facility (WSF).

The Achmmach ore contains traces of sulphides. The introduction of ore sorting will separate rock which is of low tin grade, but which may contain sulphides. This material will be returned underground as stope fill. Using an ore sorter will also reduce the total tonnage of tailings produced.

The tailings when discharged to the tailings storage facility (TSF) could be a source of acid seepage. To militate against this, the tailings will be dosed with sufficient limestone to minimise the acid producing potential. As a matter of standard practice, the TSF will be designed with a system to collect and recycle the small quantity of resultant lixiviate (seepage water) to the ore processing unit. Seepage from beneath the TSF is expected to follow natural bedrock downstream of the TSF where it will be captured in a seepage pond.

Stockpiles of ore, mine waste dumps and the TSF will be equipped with toe drains and sediment traps to prevent rainwater runoff from transporting sediment and fine rock material into the watercourses situated near the facilities. Drainage from stockpiles will be directed to the TSF or the WSF.

Impact on Natural Habitats

The layout of the project facilities has been designed to avoid areas of natural habitat (cork oak woodland) or minimize impacts on other areas of woodland (Aleppo pines) which have been planted where cork oak has been cleared. However, a 1.3 ha area of cork oak woodland and approximately 2.3 ha of Aleppo pines will be cleared. All clearing of trees will be carried out in collaboration with and with prior approval from the forestry commission (Haut Commissariat aux Eaux et Forêts).

Impact on Flora

The project facilities will be constructed on land which is currently used for seasonal grazing of animals, and which is scrub land for much of the year. This land is the type of habitat which is the least environmentally sensitive in the concession area. Cork oak woodland is the natural habitat of the area, but much of this has been cleared by local people. The construction of the facilities will therefore create a direct impact on the vegetation on this area, but it is of low environmental sensitivity.

Impact on Fauna

The change in land-use will represent a loss of habitat for some fauna which are predominantly small rodents. This impact is expected to be negligible, as these animals can move to areas nearby. The loss of habitat is not expected to have a detectable impact on birds or reptiles. The impact will take place during the construction phase and no further impact is expected during the operation of the mine.

Impact on Air Quality

The concession is in a mountainous area far from any major sources of air pollution. The baseline survey has measured the air quality and confirmed that there are no signs of air pollution. During the construction and operation of the project, dust will be generated by traffic moving along the access road. This will be minimised by controlling traffic speed and if necessary periodic spraying of sensitive areas along the road using water and a suitable dust suppressant. Blasting fumes and exhaust emissions are tightly controlled in underground mines and will not cause a detectable change in air quality outside the concession. The use of crushing and grinding equipment could create airborne dust emissions. To control emissions the installations will be fitted with covers and screens and dust suppression equipment.

Community Health and Safety

The risks with respect to community health and safety comprise

- risk of road accident
- degraded groundwater quality
- reduced water availability
- exposure to infectious diseases.

These risks will be managed through the implementation of environmental and social management plans including:

- a health and safety plan including road safety aspects
- a water use and wastewater discharge plan
- a groundwater monitoring plan.

9.6.3 Socioeconomic Concerns

The main concerns of the local people are:

- employment opportunities for local people
- working conditions (workers' rights)
- pollution and water scarcity
- compensation rights for relocation
- health impacts due to dust and water pollution
- accidents risks because of an increased road traffic.

9.6.4 Positive Socioeconomic Impacts

The project is expected to contribute to a general improvement of quality of life and health in the area. The positive social impacts include:

- Creation of direct employment. During the construction phase there will be job opportunities for around 230 skilled Moroccan workers and 120 unskilled Moroccan workers, i.e. a total of 350 people. During the operation phase there will be job opportunities for about 86 professional and supervisory staff, 245 skilled and semi-skilled personnel and about 10 unskilled Moroccan workers, i.e. a total of about 340 people. These figures have been estimated for the feasibility study and may change as the project matures.
- Payment of rent for the use of the collective lands. This will not vary in value with changes of the quality of the growing seasons as do pasture and crop yields and could offset seasons with poor yields.
- The opportunity for young people to gain employment locally rather than leave the district.
- Local companies and people will have opportunities to provide different types of services. It is estimated that contracts with local companies and peoples will represent between 25 and 35 million dirhams per year (US$2.6M to US$3.6M).

- Atlas Tin will improve sections of the access road as part of the mine construction works. This action will benefit the local people using the road.

9.7 Resettlement

The resettlement/compensation process will be as follows:

- project representatives will initiate discussions with the people concerned and an "expertise committee" will be formed that will include the residence, local commune presidents and representatives from the Governor's office.
- If an agreement is reached it will be ratified by the Ministry of the Interior (MoI).
- If an agreement cannot be reached the matter will be handled by the MoI, which will determine the level of compensation payable by the project, with the assistance of the provincial administration in El Hajeb.

The resettlement activities will be implemented with appropriate disclosure of information, consultation and informed participation of those affected. The project resettlement action framework describes the guidelines and principles related to entitlements of the affected households and the resettlement procedures. The grievance mechanism will allow displaced persons (and other members of the community) to raise concerns about the compensation and relocation procedure and outcomes. The homes that Atlas Tin are seeking to resettle are provided in Figure 9-4 to Figure 9-7 below.

Figure 9-4 Mimoun & Aziz Ait El Kabir's houses

Figure 9-5 Ittan Mohamed's house

Figure 9-6 Abid Oufdil's house

Figure 9-7 Alami Hakkou's house

9.7.1 Resettlement Compensation

Atlas Tin has continued to maintain frequent communication with all levels of local and regional government. While there is not set pricing for such housing resettlement valuation, Atlas Tin has been provided a range for compensation for house land space of between 400 Dh/m^2 (US$45/m^2) and 1000 Dh/m^2 (US$111/m^2) based on similar compensation agreements. For the basis of the resettlement and compensation cost estimate, the highest value per square metre has been assumed and although some houses are smaller than 100 m^2 an average of 300 m^2 has been assumed.

The government process for resettlement requires that all residences are to be vacated three months from the date of payment. This period may be increased during winter depending on the circumstances. Table 9-2 below outlines the costs of resettlement and compensation that has been included in the 2018 feasibility.

Table 9-2Cost of resettlement and compensation included in the study

Residence	Square meterage	Maximum value/m^2 (US$)	Value (US$)
Residence 1	300	$111	$33,300
Residence 2	300	$111	$33,300
Residence 3	300	$111	$33,300
Residence 4	300	$111	$33,300
Residence 5	300	$111	$33,300
Residence 6	300	$111	$33,300
Residence 7	300	$111	$33,300
Consulting and household support			$50,000
Total			$283,100

9.8 Waste Rock / Tailings Characterisation

Static and kinetic geochemical testing conducted by Golder Associates in 2012-2014 indicated that some of the rock types and the tailings are potentially acid forming and have low buffering capacity. Laboratory results show metals can be readily released at concentrations above Moroccan regulatory limits under neutral pH and that metal release is exacerbated when acidic conditions develop. This does not prevent the use of materials for construction of the tailings storage facility (TSF) or other site facilities, provided seepage from such facilities is managed during operation and post-closure.

Based on the kinetic leaching testing; sulphate, arsenic, copper, iron, manganese, nickel and zinc could be released at concentrations that exceed IFC and Moroccan regulatory guidelines. The addition of crushed limestone to the waste rock and tailings will reduce the potential for acid generation and metal leaching in the short-term. However, depletion calculations from kinetic testing of tailings suggest that the amount of limestone added for the test performed (3%) will not sustain acid buffering in the long term. Doubling the amount of limestone to 6% will sustain acid buffering based on preliminary calculations; this requires further testing given that mineral consumption rates will vary in time. The tailings were readily neutralised with 30 kg/t limestone fines obtained from a commercial quarry close to Achmmach. Kinetic testing of limestone amended tailings containing 30 kg/t limestone showed all components of the leachate to be within World Bank water quality standards within 26 weeks of testing.

Limestone is an inexpensive and readily available source of neutralisation potential; however, the long-term availability of buffer capacity provided by limestone is yet to be confirmed. Based on the results of the preliminary test work completed in 2012-2014, further larger scale geochemical investigation will be carried out during the final design phase.

9.9 Waste Rock Management

Waste rock generated during mine development will be stockpiled adjacent to the mine portal. The rock is expected to be mildly acidic and there is a risk of acid drainage. This will be mitigated by blending finely crushed limestone into the waste as it is placed on the stockpile. The overall quantity of excavated rock to be stockpiled during the initial development phase is estimated to be 150,000 tonnes. It is anticipated that much of this material will be used in the construction of the run of mine (ROM) pad and the TSF embankment.

Mine waste generated during the production phase will be used as fill for the expansion of the TSF embankment and the balance for backfill.

9.10 Closure Plan

The current Achmmach life of mine plan is 10 years. Although it is possible that discovery of additional resources will extend the mine life, closure is assumed to occur at the end of the initial planned life. This ensures that where appropriate the requirements for closure can be accommodated in the design. At the end of the Achmmach mine life the site will be closed with the demolition and removal of the majority of site infrastructure and associated reclamation and rehabilitation of the site.

9.10.1 Legal Obligations

There is no Moroccan legislation with respect to mine closure. The IFC requirements for mining projects with respect to mine closure are defined in the guidelines for mining:

- The Moroccan mining law requires titleholders to prepare an abandonment plan, the contents of which are to be set out in subsequent regulations. To date, no such further regulations have been published in the official journal confirming the contents of the abandonment plans.
- Article 6(4) of Moroccan law no. 12-03 on EIAs provides that the assessment must include the "measures envisaged by the applicant to remove, reduce or off-set the negative consequences of the project on the environment, as well as measures to enhance the positive impacts of the project". As the EIA is a prerequisite to the approval of the project and the applicant undertakes to implement the provisions of the EIA, the approved rehabilitation measures under the EIA will form part of the requirements for the execution of the project.

9.10.2 Closure Objectives and Outcomes

Atlas Tin has designed the Achmmach project in accordance with the underlying expectations and requirements of the IFC/Equator principles. In addition, Atlas Tin has maintained a philosophy for its design and operational phase to minimise the impact on the environment and the community.

Examples of where the principles have been applied include:

- the restriction of activities that would impact on the forestry areas
- the establishment of infrastructure on pre-disturbed and non-productive lands including the use of existing site roads and infrastructure
- minimising the operations surface footprint through innovative design and practices
- the introduction of modern technology (ore sorting) that reduces the volume of the mine tailings facility
- the selective removal and stockpiling of topsoil for future rehabilitation
- the implementation of water efficient processing design and operating practices
- reuse of waste rock from mine development as underground fill for ground support limiting the need for the permanent construction for surface waste stockpiles.

The IFC requirements for mine closure are defined in the EHS guidelines for mining (ESIA Appendix 9B). The closure plan is conceptual in nature and involves the removal or plant, blocking of the underground workings and rehabilitation of the site. Several environmental and social concerns have been identified during stakeholder consultation. These being:

- Community Health and safety: The physical presence of the buildings, waste rock dumps, tailings and process equipment are of concern in terms of risk to community health and safety.
- Risk of pollution of water resources: The TSF could represent a source of acid leaching.
- Erosion/instability: There is concern with regards to the stability of waste rock dumps and the TSF, especially with respect to run-off water, sedimentation and risk of seismic activity.

The main applicable points of the IFC requirements can be summarized as follows:

- The costs associated with mine closure and post closure activities are included in the business feasibility analysis during the feasibility studies.

- All structures are designed to remain stable and not impose a hazard to public health and safety because of physical failure or physical deterioration.
- Tailings structures will be decommissioned so that water accumulation on the surface is minimized and that any water from the surface of the structure can flow away via drains or spillways and these can accommodate maximum probable flood events.
- Spillways, drains and diversion ditches will continue be maintained after closure as they could become choked after storm events.
- Structures are designed not to erode or move from their intended location under extreme events or perpetual disruptive forces.
- Shafts and other openings will be effectively and permanently blocked from all access by the public.
- The mine and process plant are designed to protect surface water and groundwater from leaching of chemicals.

9.11 Mine Closure, Rehabilitation and Decommissioning

At a conceptual level a mine reclamation and closure plan (MRCP) has been developed to establish that the proposed mine design and operations will not prejudice the closure of the mine in compliance with all related laws and having due regard to "employees needs and the interests of the local community". The closure plan includes both physical rehabilitation and socio-economic considerations and beneficial future land use options. The MRCP is an iterative process and will continue to be updated throughout the life of the Achmmach project in conjunction with relevant legislative bodies and the local community. The establishment of the conceptual MRCP allows for the costs associated with mine closure and post closure activities to be included in the feasibility study.

9.12 Decommissioning and Rehabilitation Activities

Mine decommissioning and rehabilitation of the Achmmach mine site have been included in the project feasibility. An outline of the key activities included in the MRCP are provided below:

- Securing or blocking mine entry and exit points. This will be achieved by collapsing mine entry points or by installing secure barriers to entry.
- Development of post-mining land use and closure objectives and criteria. This will be achieved through review of initial land use practice and assessment of potential future land use in consultation with the likely land users of the future.
- Design and construction of final landforms, drainage structures and any new infrastructure required.
- Estimating, reconciling and scheduling of rehabilitation material inventories. This will be an on-going process to maintain an understanding of the likely availability of materials required for effective rehabilitation and the sources of these materials.
- Decommissioning and removal of plant and infrastructure. Residual construction materials will made safe or sealed in the underground workings.
- Landform surface treatments (ripping, selective application of topsoil, placement of materials). This work will be carried out to establish opportunity for moisture retention and effective re-vegetation.
- Completion of rehabilitation (including revegetation).
- Commence monitoring and measurement against completion criteria.

- On-going consultation with key stakeholders.
- Handover of infrastructure and finalisation of the closure monitoring and maintenance programme.

9.12.1 Processing plant

The salvageable items of the surface infrastructure will be removed and sold. All bricks and concrete will be demolished and the rubble either placed in the underground workings or used as cover on the tailings storage facility.

The residual excavations after removal of the processing plant and associated items will be safely contoured and be revegetated.

9.12.2 Mine Workings and Access

All mine openings including shafts, declines and service holes will be backfilled or capped to prevent access.

Surface contouring and drainage of the main access openings will be established to prevent water causing erosion around these areas and blend them into the overall surface contour.

9.12.3 Tailings Storage Facility (TSF)

The current closure plan for the TSF is to allow it to dry out prior to rock armouring and top soiling of the facility.

The top soil excavated and stored during the construction phase will be used to cap the TSF. The surface of the TSF will be shaped to ensure proper run off and the long term viability of the final land form. During the design phase of the TSF erosion control measures including permanent diversion drains were incorporated in the design of the TSF.

Seepage will be pumped back to the TSF or diverted into collection ponds and allowed to evaporate.

Surface and ground water quality will be monitored in accordance with the MRCP for a period to be agreed with the relevant authorities.

9.12.4 Water Storage Facility (WSF)

The fresh water storage facility has been designed as a permanent facility following discussions with the local community that has requested it be left in place to provide water for stock and agriculture purposes.

No costs of closure or reclamation have been attributed to the water storage facility.

9.12.5 Waste Dumps

The mine design is based on using waste rock as backfill to ensure the stability of the underground workings. The 2.0 Mt of waste rock from development will be supplemented by an additional 2.1 Mt of rejected rock from the ore sorter over the life of the project. This material will be temporarily stockpiled for underground back fill as well as being used for construction of mine roads, access roads and other infrastructure.

The permanent waste dump location is the site of an historical exploration shaft and waste stockpile that has surrounding surface disturbance. At closure the waste dump areas will be contoured into a final stable shape blended into the existing topography and revegetated. The old shaft will be back filled during the building of the waste stockpile to ensure the area is safe.

9.12.6 Roads and Access

The existing road infrastructure has been incorporated in the layout of new site roads and access where practicable. Atlas Tin will work with the community to agree which roads will be required to be rehabilitated.

9.12.7 Offices and Camp

The main administration building and camp areas will be removed and the sites contoured. The local communities may wish to take ownership of this infrastructure and that will be discussed with them on closure

The cost of dismantling the administration buildings and the accommodation camp will be offset by their salvage value.

9.12.8 Power lines

The main powerline into site will remain the property of ONEE, the state power company. No costs for the removal of the main powerline into site have been included.

Power distribution infrastructure within the mine site will be safely de-energised and removed.

9.12.9 Other Buildings and Equipment

In addition to the main items outlined above, minor infrastructure will either be salvaged or removed and the area rehabilitated. This includes:

- security huts
- waste water treatment plant
- explosives magazines
- salvage and laydown areas
- fencing.

9.13 Calculation and Future Financial Closure Liability

The calculation of the financial liability associated with the project has been has been estimated in accordance with the IFC/Equator principles.

The estimate involved the identification of the key closure components of the project and their surface disturbance and outlining the activities and key scopes of works required for closure. As outlined above the mine reclamation and closure plan is an iterative plan that will continue to develop over the life of the operation.

The financial closure liability associated with the project is US$3.1 million.

9.14 Permitting

9.14.1 Required Permits

The key permits required to support construction and operations are summarized in Table 9-3 along with their status as at May 2018.

Table 9-3 Project permits and approval status

Approval	Date issued	Expiry	Authority
LE 33 2912 (Mining Lease)	12/03/2010	17/01/2022	MEMDD Fes
Environmental & social (ESIA and ESMMP)	22/12/2014	22/12/2019	Ministry of Durable Development.
Decision No. 36/2014			
Collective land rental agreement	11/12/2017	10/12/2026	Ministry of Interior.
File 10776			
11/12/2017	10/12/2026	By mutual consent	Presidents of Ait Ouikhalfen and Ras Ijerri communes
Project water supply and management	19/12/2014	Open	ABHS, Fes.
ABHS/DEPRE/SDE 2433/14			
Water extraction authorization	1/12/2012	31/12/2031	ABHS Fes Decision N° 1540/12
Development of support water bores	09/02/2017	19/02/2019	ABHS, Fes.
Licence No. 243/2015			
Permis d'Occuper	17/06/2015	17/06/2018	
Renewal applied	Decision No. 02/2015		
Communes, forestry El Hajeb +WALI Fes-Meknes			
PO exploration	Under renewal process	TBA	Decision of Wali Fes- Meknes
Water storage facility 370 000m^3	Need Application + Design + Impact Study	TBA	ABHS Fes
Power transmission	Memorandum of agreement to be signed	TBA	ONEE Casa+ Meknes
Decision to Construct	Not yet applied for	Life of mine	Communes and Provincial Governor
Explosive permit	Not yet applied for	Life of mine	Communes, Provincial Governor and MEMEE

All the of the existing and pending permits are in their normal life cycle; meaning that in addition to those that are currently held, the remaining permits are progressing though the approvals process.

Water Storage Facility

Golders UK has designed the site water storage facility. The facility is scheduled to be constructed in parallel with the tailings storage facility starter wall. The application for a permit to construct a water storage facility in excess of 50,000 m^3 is continuing.

Explosives

The Achmmach project is located in an active and historical mining region. In addition, Morocco has a large established mining industry. Morocco is serviced by several of the world's largest explosives manufacturing companies.

Atlas Tin will apply for the permit to use and store explosives at the Achmmach mine site. The application is made to the Services of Explosives, a division of Technical Control and Security under the Department of Mines. The application process is well defined, and the initial application is expected to take six months. This process will begin during the detailed engineering and design phase of the project in the second half of 2018.

This will require:

- An approved mining license LE.
- The construction of approved storage facilities at site (legislated design).
- The employment of dedicated security for the site-based magazines.
- Authorising all employees that handle explosives to do so. Explosives suppliers in Morocco are authorised to provide the necessary training to the approved level of accreditation.
- Atlas Tin to develop and implement procedures and practices to ensure the safe and efficient use and tracking of all explosives at site.

Decision to Construct

This process is an approval based on the presentation and review of the feasibility study. This process is not a formalised process and is considered to be an administrative review to confirm that the project complies with all relevant laws and has all permitting and licences in place as required by law. It will confirm that the project will operate in the manner and purpose that it was designed, constructed and approved.

Power Transmission

Atlas Tin has yet to receive final approval for the power line construction. Atlas Tin remains in contact with the by Office National de l'Electricité et de l'Eau Potable (ONEE - The National Office for Electricity and Potable Water). ONEE has provided Atlas Tin the proposed corridor and capital construction cost estimates for the 44 km, 60 kV power line from the Toulal substation. The proposed power corridor transmission line route is provided (Figure 9-8). The line will be paid for by Atlas Tin, however will remain the property of ONEE. The powerline has a construction time of 12 months. The cost and construction time has been factored into the project start up schedule. Diesel generators will be used to provide power to allow mine development to commence prior to connection.

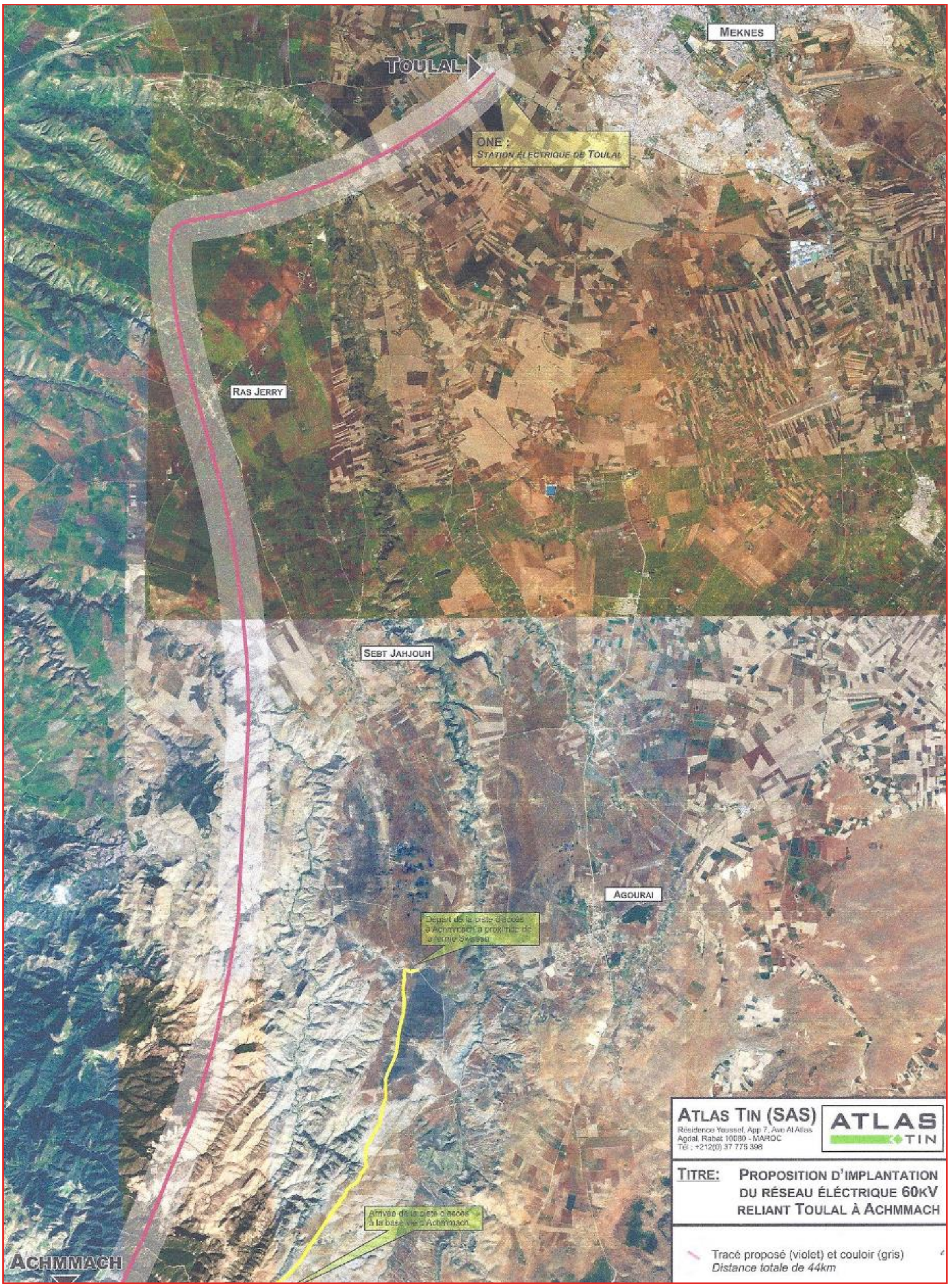


Figure 9-8 ONEE provided route from the Toulal Power Station

10 EXTERNAL RELATIONS

10.1 External Relations Program

In 2014 Atlas Tin undertook detailed and structured stakeholder engagement activities prior to and during the ESIA (environmental and social impact assessment) process. Atlas Tin acknowledges that it requires both a legal and social licence to operate and company management and the director of external affairs have continued to maintain structured and frequent contact with the stakeholders.

10.2 Activities Performed Prior To Performance Of ESIA

The stakeholder engagement activities carried out prior to the performance of the ESIA comprise both formal meetings with representatives of the government and informal encounters with local residents of the project area:

Formal meetings were held with the following groups and individuals:

- the Ministry of Energy, Mines, Water and the Environment in relation to the leasing of the Achmmach concession;
- the chairman of the National Committee for Environmental Impact Assessment with respect to the environmental permitting process;
- the Wāli of Meknès-Tafilalet region to discuss the public consultation process *(a Wāli is similar to a governor*);
- the Regional Director of the Ministry of Energy, Mines, Energy and the Environment to discuss public consultation accompanied by the ESIA consultant;
- the Director of the Centre for Regional Investment in Meknès;
- the Governor of El Hajeb province;
- the commander of the local gendarmerie (police) in Sebt Jahjouh and Agourai.

Atlas Tin employees have maintained frequent contact with the presidents of the communes where the project is situated and through which the access road passes. Core topics in the meetings have been the condition of the project access road, which is a public road and is maintained by the communes and employment opportunities for residents.

Informal encounters have frequently occurred between the Atlas Tin employees at the project site and residents in the project area. During these encounters information about the project has been exchanged. The local residents are aware of the project and have not indicated any negative sentiment toward it.

The IFC also performed interviews with the following stakeholders:

- the Regional Wāli
- the Prefect of Meknès
- the Centre for Regional Investment (CRI)
- the local mining authority and environmental authority “Office National Des Hydrocarbures et Mines (ONHYM)”
- the Department of the Environment.

10.3 Activities Performed as Part of the ESIA Process

The ESIA process started with the preparation of a scoping study that was presented to the National Committee for Environmental Impact Studies (CNEIE) on 15 June 2011.

Stakeholders consulted as part of the scoping study included:

- the Centre Régional D"Investissement, Meknès on 12 April 2011
- the Haut Commisssariat aux Eaux et Forêts et à la lutte contre la Désertification on 14 April 2011
- the Direction de l"Equipement, service infrastructure on 14 April, 2011
- the Office National de l"Electricité, Meknès on 14 April 2011
- the Office National de l"Eau Potable, Meknès, on 14 and 15 April 2011
- the Inspection Régionale de l"Habitat, de l"Urbaniseme et de l"aménagement de l"espace, Meknès on the 14 April, 2011
- the Agence Urbaine de Meknès on 15 April, 2011
- the Agence de Bassin Hydraulique de Sebou (ABHS), Fès on 20 September 2010
- the Office National Des Hydrocarbures et Mines, Meknès on 17 September 2010.

The National Committee for Environmental Impact Assessment has approved the scoping report and issued the Terms of Reference for the ESIA report.

For the needs of the social baseline study, a part of the ESIA, individual household consultations were carried out in the project area in April 2013. More than 30 households were visited and interviewed through structured and semi-structured questionnaires. Social, economic and cultural issues were discussed, as well as concerns, expectations and perceptions about the project.

Commencing in July 2013, public participation was conducted, in line with Moroccan Decree n° 2-04-564 regarding public involvement. This requires that an ESIA summary document be presented in a public hearing, and that the findings of the public hearing be considered by the National Committee for Environmental Impact Assessment when approving the report.

Post the completion of the ESIA, Atlas Tin has continued to maintain strong community and government interaction. The director of external affairs has ongoing communications with all the key stakeholders and local commune presidents. In addition, the executive management of Kasbah Resources has continued to meet and update stakeholders whilst in Morocco.

10.4 Project Stakeholders

10.4.1 Introduction

Stakeholders are persons or groups who are directly or indirectly affected by a project, as well as those who may have interests in a project and/or the ability to influence its outcome, either positively or negatively. Stakeholders may include locally affected communities or individuals and their formal and informal representatives, national or local government authorities, politicians, religious leaders, civil society organisations and groups with special interests, the academic community, or other businesses.

10.4.2 Project Stakeholders

The project stakeholders are listed in Table 10-1 to Table 10-3 below.

Table 10-1 Stakeholders directly and indirectly affected by the project

Stakeholder Group	Comments	Interest
A1. Landowners and landowner family members	There is no private land in the area of influence. All land is either forestry land or communal land with the exception of the fluorite mine 8 km away.	Not applicable
A2. Farmers who live on the concession area and who cultivate crops and graze sheep, goats and cattle.	There are approximately 50 households living on the concession area	Resettlement Compensation Environmental impact Employment and other economic activities Community benefits
A3. Communities that are hydraulically downstream of the mine	No settlements and limited land occupation between the mine and the confluence between Oued Akrid and Oued Beht	Environmental impact Employment and other economic activities Community benefits
A4. Communities that are situated along the access road	Settlements (douars) along the access road	Environmental impact Employment and other economic activities Community benefits
A6. Administrations of Morocco	Ministry of Energy, Mines, Water and the Environment Ministry of Agriculture and Maritime Fisheries Ministry of Interior	Economic development Regulates project
A7. Regional and local administrations (which comprise the Public Enquiry Committee)	Wāli of Meknès Governor of El Hajeb Province Governor of Khemisset Province Commune presidents Regional Director of the ministry of Energy Mines, Water and the Environment	Environmental impact Economic development Regulates project
A8. Other ministries, agencies and government organisations	River basin agency High commission for forest and water ONHYM Centre Régional D'investissement	Environmental responsibility
A9. Neighbouring industrial activities	SAMINE (operator of neighbouring fluorite mine)	Water resources Access road

Table 10-2 Stakeholders that have "interests" in the project or parent company

Stakeholder Group	Comments	Interest
B1. NGOs, political and community organisations operating in Morocco	Not known to the proponent at this stage	Environmental and social responsibility Community development opportunities
B2. International NGOs with general interest in mining projects in developing countries		
B3. Province community and media	Moroccan press is already aware of the project	

Table 10-3 Stakeholders the have the potential to influence project outcomes

Stakeholder Group	Comments	Interest
Administrations	See A6 above	See A6 above
Regional and local administrations (which comprises the Public Enquiry Committee)	See A7 above	See A7 above
Other ministries and government organisations	See A8 above	See A8 above

10.5 Stakeholder Engagement

As part of the stakeholder engagement process the following documents were presented to all identified stakeholders:

- 2007 Achmmach environmental baseline survey summary report
- the company's environmental and social action plan
- a public consultation document in compliance with Morocco's laws on public consultation (project overview, summary of environmental impacts, maps)
- the ESIA executive summary.

The communication methods used for stakeholder engagement were as follows:

- meetings with regulatory bodies
- meetings with local stakeholder representatives
- public exhibition of public consultation documents
- announcements in local media
- provision of general information on notice-boards at key public locations
- posting of ESIA executive summary on the IFC and Kasbah Resources websites.

The ESIA comprehensively outlines the key action plans and structure to Atlas Tin's stakeholder engagement as summarised below:

- The Kasbah executive management team will have the oversight for the programme, which will be implemented by the site general manager and the community liaison officer.
- The community liaison officer will report to the site general manager who in turn reports to the chief executive officer. The company's stakeholder engagement strategy will be communicated within the company by the issue of a memorandum by the chief executive officer and addressed to all the managers and senior staff.
- All documents produced or received in relation to the stakeholder engagement (including the grievance register) will be filed in accordance with the company's document control system. A grievance register will be established and maintained by the community liaison officer.
- A log and procedure for tracking all incoming and outgoing communications regarding stakeholder engagement will be established and maintained by the community liaison officer
- Atlas Tin has continued to maintain its stakeholder engagement activities highlighting the company's ongoing commitment to all stakeholders. Through Atlas Tin's director of external affairs and the company's executive management frequent and routine interactions both socially and formally have been maintained at all levels of the government and the local communities.

- The local commune leaders are provided ongoing updates on the project status, while Atlas Tin continues to employ people from the local communities on both a permanent and casual basis to support the ongoing camp and feasibility activities.

10.5.1 Perceptions, Concerns and Expectations of Local Stakeholders

The interviews and survey of households carried out in the study area, inside the concession and in its surrounding areas, enabled analysis regarding the interaction of the project with its environment. It is widely accepted that beyond material resources (income, access to basic infrastructure and living conditions), social factors (education level, gender), geographical factors (distance from the project site), and cultural and demographic factors (size of the household) have an influence on the households perception and understanding of the project. Thus, whilst certain expectations or concerns are shared by the whole population in the study area, others are more specific to certain social groups.

Most of the people who were surveyed do not know the name of the company developing the Achmmach project, i.e. Kasbah Resources, but most them refer to “the Australians” or other nationalities (e.g.: “the Turks”, since a Turkish company was the drilling contractor for Kasbah Resources at the time of the survey).

The Samine fluorite mine, which is near Achmmach, has influenced the expectations and perceptions of the population in the study zone. Indeed, the population sees mining projects as opportunities for local development and a way to take the area out of its current isolation. The main expectations concern employment and development of infrastructure.

The district councillors have insisted on a specific definition of local employment. They feel that, between two people with equivalent skills, priority will be given to inhabitants of the three districts of Ait Ouikhalfen, Sebt Jahjouh and Ras Ijerri. Rural populations also fear that the project will only employ people who do not live in the village and that they will therefore not benefit from any potential windfall. They hope that a fair and transparent recruitment process will be implemented, thereby giving local populations access to available jobs.

The distribution of benefits in terms of employment was a recurring topic: if several members of a given family were to be recruited when other families did not have access to any jobs created by the project, frustration and tension could build up between the families. None of the women expressed interest in working at the mine. They consider that the jobs on offer are exclusively for men and people who are more educated than them.

The issue of infrastructure, particularly roads, but also health care and education facilities, represents the second most important topic for local populations and councillors. To ensure people do not have unrealistic expectations, the project team will regularly inform the population and its representatives of project progress and matters which may affect the community.

Other concerns related to pollution risks, and the possibility of pollution of the water resources in particular. The exhaustion of water resources is a concern that is shared by the local population and its representatives on a district level.

Others are wary of the consequences of the increase in traffic and the flow of construction equipment, including accident risks and potential nuisances (like dust) that could be induced by project related activities.

Lastly, local councillors expressed their views on staff housing. They expect the employees would be lodged in the existing urban centres, such as Sebt Jahjouh, Ras Ijerri and Agourai. This option would provide local development opportunities and would avoid abandoning the staff residence zones when the project cycle has been completed.

Atlas Tin shares the view of the local councillors and intends to transport employees to the site by bus from regional and local centres where they live. There will only be a small number of expatriate staff and a few Moroccan staff housed at site.

10.6 Resources and Responsibilities

The key company resources for the implementation of the stakeholder engagement plan (SEP) comprise:

- the executive management team represented by the Chief Executive Officer
- site General Manager
- Community Liaison Officer.

The company will engage an ESIA consultant for assistance.

10.6.1 Executive Management Team

The executive management team represented by the Chief Executive Officer has the overall responsibility for stakeholder engagement. Its role is to provide the guiding principles for the stakeholder engagement, allocate budget for the effective implementation of the SEP and if necessary participate in the dialogue with administrations in the resolution of any issues.

10.6.2 Site General Manager

The site General Manager will be responsible for the implementation of the SEP, which includes overseeing the work carried out by the community liaison officer and the ESIA consultant. The position is the point of contact with the regional and local authorities with respect to public consultation.

10.6.3 Community Liaison Officer

An Atlas Tin staff member has been appointed as the Community Liaison Officer (CLO). The officer is a native Moroccan and familiar with the local communities and will act on behalf of the chief executive officer and report to the site general manager. The CLO's specific responsibilities are to:

- address any grievances expressed by the local communities through the grievance mechanism
- identify any significant new issues that may arise as the project progresses
- assist the chief executive officer and site general manager and local communities to resolve any disputes between the parties
- maintain records relating to the consultations, and collation and preparation of internal and external reports.

10.7 Grievance Mechanism

The ESIA outlines the grievance process that will be adopted by Atlas Tin. Listed below is a brief description of the grievance process that has been adopted:

Step 1 – Receipt of grievance: Grievances will be received either verbally or through written notification to the CLO at the site of operations. Grievances received by other project employees will be forwarded to the CLO. Atlas Tin will provide different means that will allow the public to report concerns or complaints, e.g. through regular meetings with the affected communities and their representatives (commune presidents).

Step 2 – Review of the complaint: The CLO will review the complaint to determine severity of the grievance and how to proceed. Depending on the nature of the complaint, the CLO will seek management assistance, commence an investigation or resolve the issue through direct interaction with the complainant.

Step 3 – Document the complaint: All the complaints will be recorded and kept in a communications/ complaints log book. The record will contain the name of the individual or organisation making the complaint, the date and a description of the complaint, any follow-up actions taken, the final result, and how and when the decision was communicated to the complainant.

Step 4 – Response time: All communications will be acknowledged within 5 business days and a response provided within 30 business days. A communications procedure and log will be developed for use during the period of the SEP. This will be used to log all significant incoming and outgoing communications.

Step 5-Investigation and grievance resolution: Atlas Tin will, where necessary, conduct an internal investigation to determine the underlying cause of the grievance and will make any changes required to internal systems to prevent reoccurrence of a similar grievance. Atlas Tin will hold meetings with the person/group expressing the grievances to discuss, clarify and solve the issue, and prevent it from reoccurring.

Step 6 – Close-out: Once the investigation is completed, the results will be communicated to the complainant.

Step 7-Follow-up: The CLO will follow-up with the complainant to ensure that the grievance has been addressed in a satisfactory manner.

10.7.1 Monitoring and Reporting

Monitoring

Monitoring during the ESIA disclosure phase comprised coordination with the regional public consultation committee.

During the construction and operation phase, monitoring will comprise:

- Monitoring of environmental and social performance indicators defined in the ESIA (chapter 6, treating of monitoring). The Law 12-03 for environmental impact assessment requires a monitoring programme with indicators and frequency for monitoring. The monitoring will be carried out by the CLO.
- There will be monitoring of concerns and grievances made by stakeholders through the grievance mechanism. All grievances will be recorded in a grievance register.
- Monitoring of concerns or issues expressed by the Ministry of Energy, Mines, Water and the Environment, the Wāli of Meknès or the Regional Public Consultation Committee will be undertaken by the site general manager.

Reporting

Reporting during the ESIA disclosure phase comprises following the Moroccan procedures for public consultation.

During the construction and operation of the mine the CLO will produce a monthly report detailing the monitoring of the performance indicators and any concerns regarding grievances. The monthly report will be sent to the executive management team.

The site general manager will report to the executive management team matters raised by any of the following:

- the Ministry of Energy, Mines, Water and the Environment
- the Wāli of Meknès
- the Regional Public Consultation Committee
- the National ESIA Committee.

The company quarterly and annual reports will include a summary of stakeholder engagement activities.

10.7.2 Contractor Management

The effective management of contractors with respect to residents and the maintaining of good relations will be ensured through the following actions:

- preparation of a good practice charter for contractors regarding the respect of local people and communities
- making the respect of the good practice charter a contractual obligation
- briefing of company staff on the requirement and importance that contractors comply with the good practice charter.

10.8 Regional Industry and Infrastructure

The Project is located in a region of historic and ongoing underground mining operations. Morocco hosts an established modern mining industry and associated support services industries, primarily associated with phosphate mining operations.

Expatriate employees will be housed at the mine site camp while the majority of employees will come from the city of Meknès and the surrounding communities.

The city of Meknès is a major regional commercial and government centre. Meknès hosts modern, industry infrastructure and support that will support the Achmmach project. Meknès also provides access to modern educational institutions including universities, modern medical facilities, accommodation and housing. Meknès is also a popular tourist destination.

The project is not considered to be in a remote location. The administrative region of Meknes – Fez is located 55 km from the project and is the third largest city in Morocco with a combined population of ~ 1.55 million (Meknes pop ~ 600,000 and Fez pop ~900,000). Rabat the Moroccan capital is 150 km from Meknes and the second largest city in Morocco with a population of 1.6 million. While the largest city in Morocco, Casablanca in 246 km from Meknes and has a population of 3.2 million people. The modern transport and road infrastructure of highways, rail and regional and international airports means that the Achmmach project is ideally positioned and extremely well serviced.

11 RISK MANAGEMENT

11.1 Project Risk Review

A project risk workshop was undertaken in May 2018 with key corporate project personnel to identify, analyse and propose control measures to ensure achievement of the project objectives. This process took into consideration key risks at the following phases of the project – development, approvals, execution, commissioning/ramp up, steady state operations and closure, but uncontrollable risks such as funding and tin price are specifically excluded. Controls to manage key risks will be incorporated into the project implementation plan. Through the successful implementation of the proposed risk management plans it is expected that the project will safely and efficiently achieve its objectives.

11.2 Risk Analysis Outcomes

This section provides a summary of the key project risks and their controls. For an overall summary of the assessed risks refer to Table 11-1.

11.2.1 Mining

The risk of "*delay/poor performance of the mining contractor*" has the potential to delay the critical path and defer revenue generation. To minimise this risk a comprehensive due diligence process will be conducted on the selected mining contractor prior to award of the contract. The contract will have schedule completion targets to ensure the required productivity levels are maintained throughout the term of the contract. To provide an incentive to meet the schedule, payment of a proportion of the contract value will be contingent on the contractor meeting agreed schedule milestone. Safety will also be factor in determining the inventive payment to prevent shortcuts being taken to meet the schedule.

While unlikely, there is a risk of "*higher than expected ROM variability (tons and grade)*" that could lead to a delay in achieving the scheduled ore production, and lead to a short fall in forecast tons and/or grade. These are factors relating to the geology of the deposit which are beyond the control of the contractor.

The planned grade control drilling program will provide the necessary data for modelling and analysis to increase orebody certainty. In addition to the improved resource knowledge it is intended that the mine schedule will target several areas to be mined concurrently. This will lead to a detailed and robust mine plan with sufficient contingency or redundancy to manage remaining orebody variability. The current geotechnical analysis indicates that there are no significant geotechnical characteristics that will adversely impact mining operations.

11.2.2 Processing

Whist unlikely, there is the risk that a "*water imbalance*" could result in additional capital investment to secure required water supply for the process plant. This risk would be as a result of the process plant utilising larger volumes of water than projected and has been stored in the water storage facility.

Several measures are planned to ensure a sustainable water supply, including the early construction of the water storage facility to capture water reserves prior to start up, and the process flow sheet producing tailings as paste to minimise water consumption.

It is possible that the process plant will not achieve planned throughput and/or recovery which could lead to a short fall in tin concentrate production during the initial stages of operation. To date, bulk test work has been completed and validated by an independent third-party consultant to give confidence in the process design. The project will focus on increasing certainty around plant performance, by focussing on:

- commissioning, and ramp up plans
- maintenance strategies
- equipment and control systems selection that will maximise plant availability
- spares inventory and
- recruitment and training of process plant operators.

In addition, the engineering contractor will have contractual obligations to meet the design specification of the plant in terms of throughput, run time, concentrate specs etc; thus transferring the risk from Atlas Tin.

11.2.3 Human Resources

The inability to secure a suitability skilled and motivated workforce has the potential to affect productivity which could delay the critical path. The project strategy is to use a local workforce to support construction and operations. To achieve this, several resourcing related controls will be put in place, including:

- early engagement with local communities to identify potential employees and their roles
- development and implementation of a recruitment and retention plan
- resourcing and establishing a training plan to build greater knowledge and competency.

The inability to attract and retain key technical roles within the project whilst a risk, is not the same magnitude as that presented by the attraction of the general workforce. The project strategy is based on utilising expatriate labour for technical roles during the early phases of project, and then transition to local personnel to fill these roles. Technical roles will be engaged during the construction period to ensure capability and capacity is established in the early stages of this project.

11.2.4 Health, Safety, Environment & Community

An "*uncontrolled release of tailings*" from the TSF (tailings storage facility) is not expected to result in any significant safety or environmental impacts, so has low risk. The tailings chemistry is relatively inert, and the TSF is not upstream of any local communities or areas of environmental significance. Therefore, the overall impact of this risk is largely contained to the economic impact. Nonetheless, there has been rigorous design, construction and quality assurance processes undertaken and which will be completed prior to the facility being deemed fit for operations. The risk of TSF failure is considered rare. The TSF will also have a detailed operating plan to ensure monitoring and deposition practices are actively managed.

It is unlikely that the project will be subject to a "*serious security, safety, health or hygiene incident*" with the proposed level of control being applied to the project. Key controls include:

- commitment to ensuring ongoing adherence to the IFC/Equator Principles
- establishing a travel management plan for the transfer of project workforce (air/land)
- establishing appropriate security arrangements to protect project personnel and assets based on a specific risk assessment addressing these matters

- developing an integrated project safety management system, and associated training and workforce development; and
- ensuring any supply chain partners will adhere to Atlas Tin's health, safety, environment and community policies.

11.2.5 Capital Expenditure

The risk of "*unexpected growth in the capital estimate*" is considered unlikely as there has been detailed, third party verified, engineering and capital estimate for the processing plant and infrastructure. The mine which includes substantial capital development in the initial years is costed based on pricing submissions from three mining contractors. The estimates will also be supported by a contingency allowance and contracting strategy that appropriately allocates and manages risk.

Due to the dry and relatively predicable climate, the risk of weather delays to the construction schedule is minor. However, the detailed project construction schedule includes appropriate delays based on historical weather data.

11.2.6 Ownership & Legal

There are a small number of homes located in the project area which will be relocated. The risk of a delay in resettlement of community members is relatively low, due to the active engagement of the affected residents, who have not indicated any reluctance to move, and the support of a government resettlement process.

It is considered unlikely that a delay to completing permitting will have a material effect on the project critical path. The project already has in place most of the key approvals, including the Licence to Exploit, and only requires approval of explosives, powerline and water permits, none of which are on the critical path or have an onerous process. Atlas Tin continues to engage with relevant government departments to ensure a common understanding of the project's progress, and what is required to obtain and maintain the necessary approvals. The project has engaged with the state's electricity provider to supply electrical power and negotiations are underway to ensure the timely connection. Portable generators can be mobilised to ensure works commence as scheduled in the event there is a delay in connection.

11.2.7 External Relations

The project's continued success depends on maintaining the support of the local community and government. The project is committed to ensuring the ongoing support of key stakeholder groups, through the demonstration of key controls, including:

- commitment to the IFC/Equator Principles
- adherence to Atlas Tin's corporate governance policies and procedures, including third parties involved in the project
- application of a stakeholder management plan, which considers the broad range of relevant stakeholders and their expectations
- the development and application of a community engagement strategy that includes the appointment of key local representatives.

11.3 Project Risk Matrix

Table 11-1 is the project risk matrix and represents the assessment of key project risks, taking into consideration the existence of current and planned controls. As the project is implemented, controls will be applied that reduce these risks to an acceptable level.

Table 11-1 Project risk matrix

		Consequence Rating				
		1-Insignificant	**2 -Minor**	**3 -Moderate**	**4-Major**	**5-Catastrophic**
Likelihood Rating	**E-Almost Certain**					
	D-Likely					
	C -Possible		5-Weather adversely effects construction 21-Delay/ inability to negotiate a relocation settlement with local communities		40-Delay/poor performance of the mining contractor	34-Higher than expected ROM variability (tons and grade) 35-Lower than forecast metal tonnes (throughput/ recovery)
	B -Unlikely			39-Water imbalance 30-Inability to attract and retain required technical roles	3-Delay to obtaining required approvals 6-Unexpected growth in the capital estimate 31-Inability to attract and retain the required workforce	15-Serious security, safety, health or hygiene incident
	A-Rare				19-Uncontrolled release of tailings	20-Loss of support with key stakeholders

LOW	MEDIUM	HIGH	EXTREME

11.4 Risk Review Process

The risk review process was facilitated by MYR Consulting (risk consultants) and involved the following steps outlined in Figure 11-1.

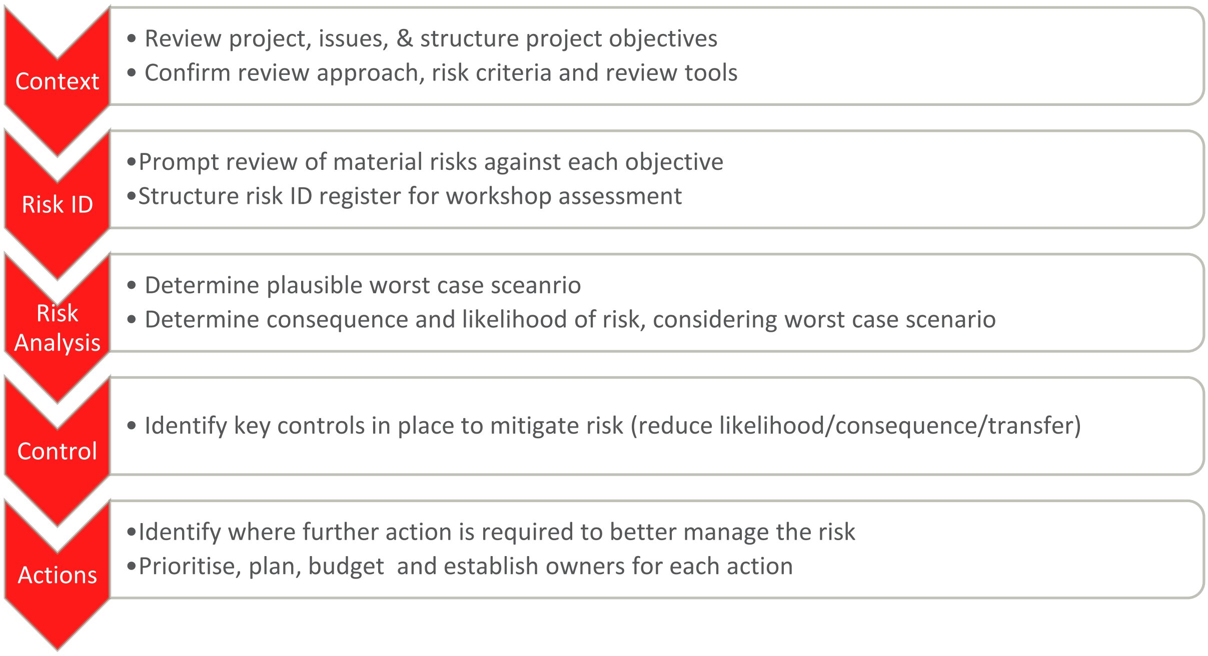


Figure 11-1 Risk review process

The project objectives listed in Table 11-2 were considered when identifying relevant material project risks.

Table 11-2 Project objectives

ID	Subject	Objective
1	Schedule	To fulfil key project milestones
2	Capex	To deliver the project within the current estimates
3	Viability	To demonstrate the financial and technical viability of the project to satisfy internal requirements, and to maintain support from relevant stakeholders
4	Occupational Health and Safety	To deliver and operate the project without incident
5	Environment,	To deliver and operate the project with minimal adverse impacts to the natural environment
6	Community	To ensure relevant communities maintain their ongoing support for the project
7	Operational Requirements	To establish systems, infrastructure and information required to ensure the ongoing safe operation of the project
8	Ramp up & Steady State	To safely ramp up to the required throughput and concentrate quality measures
9	Steady State	To achieve required throughput rates, quality measures and operating costs at steady state operations

11.5 Risk Assessment Tables

The standards and criteria used to assess project risks are summarised in Table 11-3, Table 11-4 and Table 11-5.

Table 11-3 Risk consequence

	1. LOW	2. MINOR	3. MODERATE	4. MAJOR	5. CATASTROPHIC
Health & Safety	First aid treatment required	Medically treated injury/ illness	Restricted duties/short term lost time injury/illness	Serious lost time injury/ illness	Fatality or permanent disabling injury/ illness
Financial (US$)	<$50k	$50k-$500k	$500k – $5M	$5M-$25M	>$25M
Environment	Confined impact to low significance area easily remediated. (i.e. single animal death, 20 – 250l oil/chemical spill, 1 bag of caustic)	Confined reversible impact on environment. (i.e. some wild life deaths, 250l – 1000l oil/chemical spill)	Local area onsite/offsite reversible impact (<1yrs) remediation. (i.e. large number of animal/plant deaths)	Onsite/offsite reversible impact (>1 yrs) remediation. Extensive plant/animal deaths affecting ecosystem function	Irreversible widespread damage to plant/animal populations
Compliance	Non-conformance with internal procedure. No regulatory reporting	Non-compliance with external standard/ contract or procedure. Reportable to regulator, with low potential for impact.	Non-compliance with external legal requirement. Reportable to regulator, potential fines/penalties/ formal censure	Single significant breach of laws resulting in prosecution. Failure to meet standard audit.	Extended suspension/ loss of license to operate
Reputation	Public concern restricted to local complaints.	Adverse attention/ complaints from local public/ media/ investors	Adverse attention/ complaints from local state media/NGOs/Investors	Ongoing adverse attention/ complaints from national media / NGOs/ Investors	Strong adverse attention/ complaints from international media/NGOs/Investors
Community	Low-level repairable damage to relationship	Expression of mistrust with some impact on decision makers/public opinion	Ongoing social issues impacting public opinion/ decision makers.	Ongoing social issues. Prolonged period to repair damage to relationship. Damage to item of cultural significance	Widespread ongoing loss of trust across the community. Irreparable damage to highly valued item

Table 11-4 Likelihood rating

Descriptor	Description
A-Almost Certain	Happens from time to time and could happen in next (three months), 85% chance of occurring
B-Likely	Has happened in the past and could happen in next (six months), 65 to 85% chance of occurring
C-Possible	May occur within (one) year, 40 to 65% chance of occurring
D-Unlikely	Not likely to occur but may occur in next (three) years, 15 to 40% chance of occurring
E-Rare	Only likely in exceptional circumstances and unlikely to occur in next five years, less than 15% chance of occurring

Table 11-5 Risk matrix

	1 – INSIGNIFICANT	2 – MINOR	3-MODERATE	4-MAJOR	5-CATASTROPHIC
A-Almost Certain	A1-MEDIUM	A2-HIGH	A3-HIGH	A4-EXTREME	A5-EXTREME
B-Likely	B1-MEDIUM	B2-MEDIUM	B3-HIGH	B4-HIGH	B5-EXTREME
C-Possible	C1-LOW	C2-MEDIUM	C3-HIGH	C4-HIGH	C5-EXTREME
D-Unlikely	D1-LOW	D2-LOW	D3-MEDIUM	D4-MEDIUM	D5-HIGH
E-Rare	E1-LOW	E2-LOW	E3-MEDIUM	E4-MEDIUM	C3-HIGH

11.6 Risk Management

11.6.1 Operational Risks

Atlas Tin will develop a risk management plan for construction and mining operations which identifies the mine site hazards using a formal identification audit process and establishes the risks that could arise from those hazards, including for each risk:

- the nature of the risk
- the likelihood of the risk arising
- the likely consequences if it does arise
- the control measures to be implemented
- the efficacy of control measures in the event of an occurrence
- the specific actions required in the emergency response plan if applicable.

The hazard identification will address the following functional areas within the mine.

- health and safety
- reagents, fuel and explosives
- electrical
- plant and mechanical
- environmental
- site security

The risk management process will be the responsibility of the EHS manager who will report monthly to the general manager on aspects of the risk management process. Reports will include details of any changes in high or extreme risks and document progress of the periodic revision of the site wide risk audits.

11.6.2 Commercial Risks

Risks to business continuity in addition to the operational risks discussed above include commercial matters such as compliance and financial risks. The Chief Operating Officer and the General Manager will implement a process for identifying, assessing internal and external commercial risk factors. This will include a schedule for the routine auditing of business areas to assess changes which may lead to a variation in the risk profile.

11.7 Risk Reporting Requirements

The General Manager will provide a monthly operations report to the Joint Venture Committee which will include inter alia:

- details of hazard identification audits and risk assessments completed
- details of new or changed risks that are rated as high or extreme
- details of mitigation strategies
- assessment of the efficacy of mitigation strategies when implemented.

The mine risk reporting process will be aligned with the corporate risk reporting requirements of the joint venture partners.

12 OWNERSHIP, LEGAL, AND CONTRACTUAL

12.1 Ownership

The concession area for the project straddles the border of the two neighbouring provinces, Khemisset and El Hajeb and lies within the rural districts of Ras Ijerri, Sebt Jahjouh and Ait Ouikhalfen.

Atlas Tin SAS (Atlas Tin) is a joint venture company owned by:

- Kasbah Resources Ltd (75%)
- Toyota Tsusho Corporation (20%)
- Nittetsu Mining Co Limited (5%).

Kasbah Resources Ltd is the manager of Atlas Tin under a management services agreement. The details of the joint venture arrangements are presented in Appendix 12A.

12.1.1 Access to Property

Morocco has well developed national infrastructure including rail, road, sea ports and airports. Access to the project is from Rabat, west along the A2 expressway via Meknes for 150 km then 35 km south along a sealed road to Agourai, and a further 20 km south east along an unsealed rural road to the project site as shown in Figure 12-1.

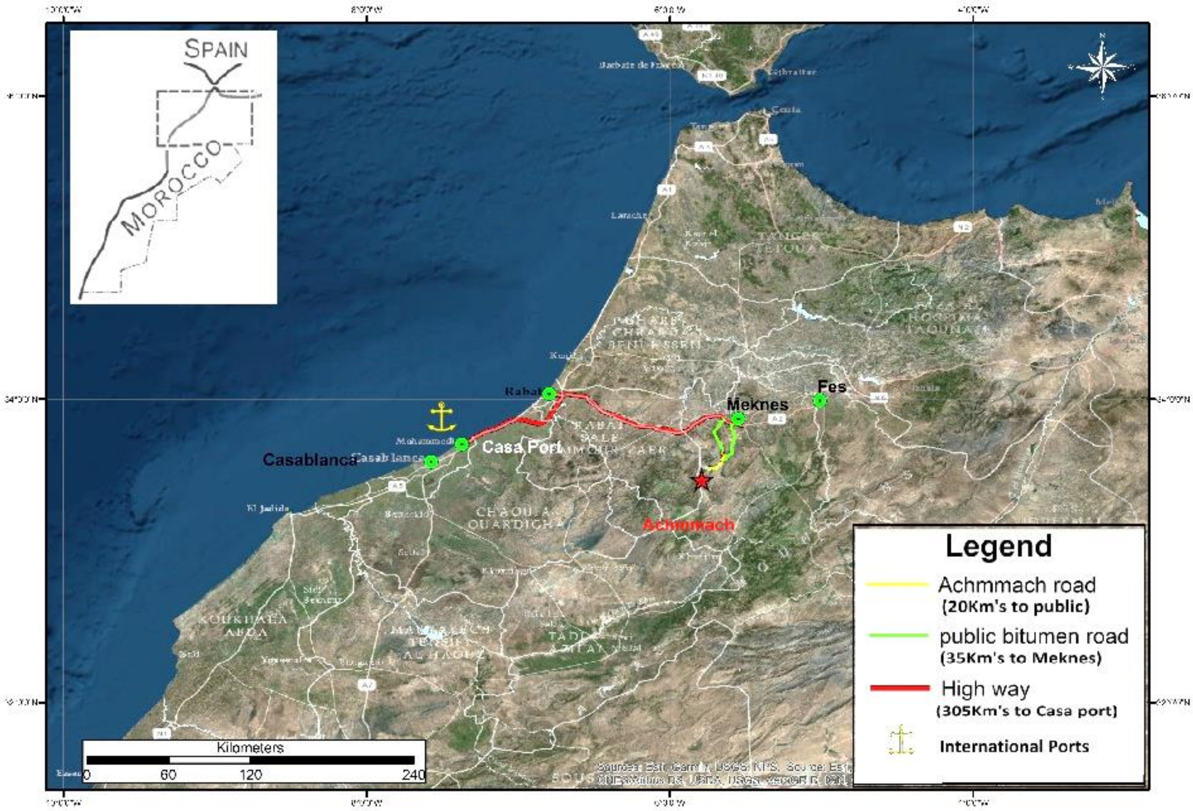


Figure 12-1 Site access

12.2 Tenure Agreements

12.2.1 Tenure

In May 2016 the Moroccan government introduced a new law on mining in which required all mining and exploration companies to convert their existing licenses to the new standard. In August 2017, Atlas Tin was granted its operating licence (licence d'exploitation) LE 332912 (Figure 12-2) in accordance with the new law on mining with an expiry date of 17 January 2022. The licence covers an area of 11.9 square kilometres.

The new operating licence is granted for ten years and renewable for periods of ten years until reserves are exhausted. Under the new law, the holder of the mining operating licence has to be a Moroccan legal entity.

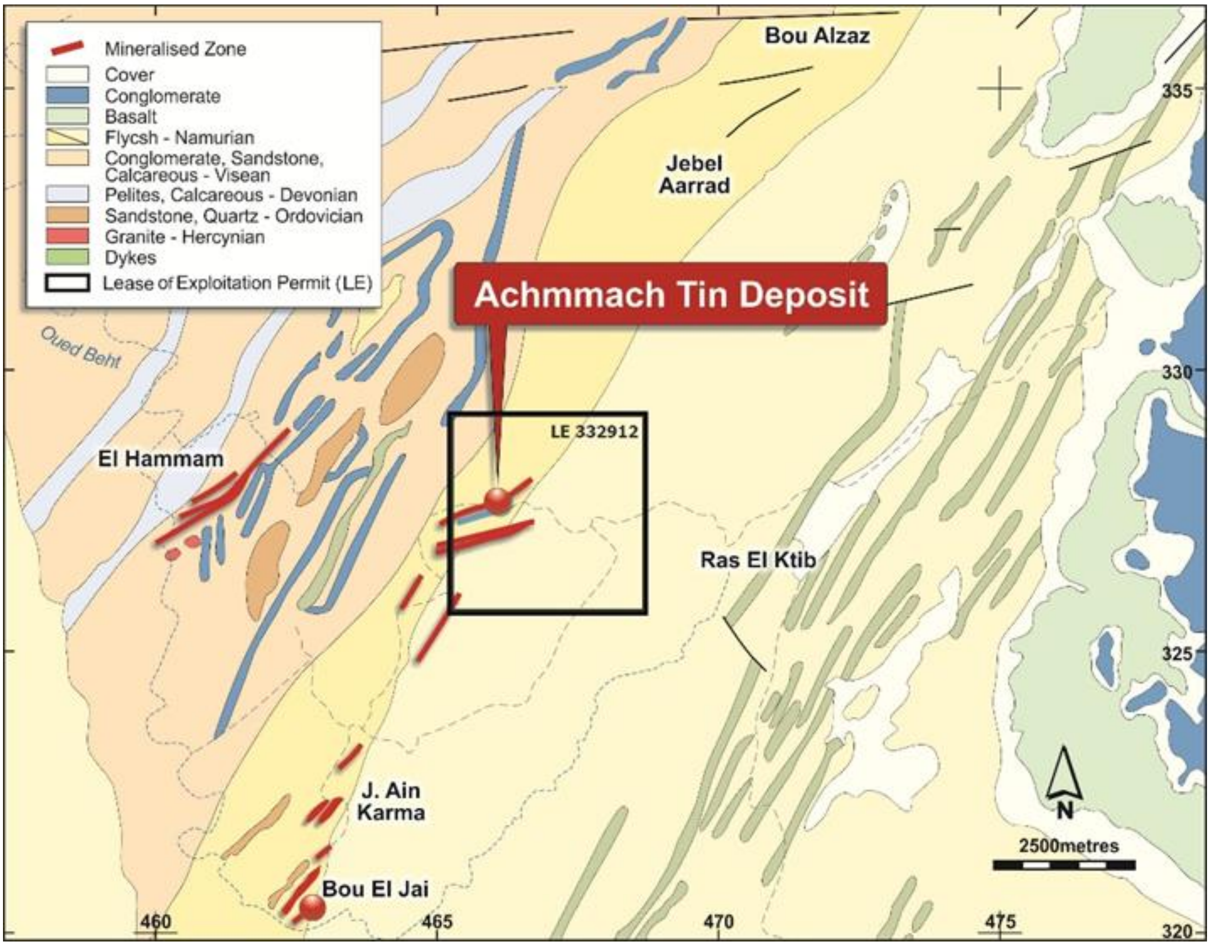


Figure 12-2 Achmmach geology and exploitation permit

12.2.2 Surface Rights

The operating licence LE 332912 straddles two communes: Ait Ouikhalfen and Ras Ijerri. There are approximately 300 people living in the concession area and five households are located in the immediate vicinity of the future mining facilities. These five households which comprise seven families will be resettled. The land tenure system in Morocco is comprised of: state owned lands, collective lands and private lands locally known as “melk”. Most of the lands in the project area are collective lands. These are administered by a Jmaâ (assembly) composed of Naibs (representatives of the land right-holders), under the direct supervision of the Ministry of the Interior (through the Directorate of Rural Affairs – Direction des affaires rurales).

The project site occupies both cleared collective land and uncleared forestry land. Moroccan law states that neither of these lands can be sold. However, long term rental agreements overseen by the Ministry of the Interior have been formalised. Table 12-1 below outlines the current state of land agreements.

There are no archaeological or formal cultural sites in the project area according to the EISA. However, there are a few spiritual sites in the concession area and its vicinity, among which is the Sidi Addi peak, which has spiritual significance for the local peoples as it is considered as the mausoleum of the Saint Addi.

Table 12-1 Land agreements

Approval	Date of Issue	Relevant Authority
Environmental and social (ESIA and ESMMP)	22/12/2014	Ministry of Environment Decision No. 36/2014
Collective land rental agreement	25/11/2014	Ministry of Interior File 10 776
Community development agreements	17/10/2014	Presidents of Ait Ouikhalfen and Ras Ijerri communes
Project water supply and management	19/12/2014	Sebou Basin Agency, Fes. ABHS/DEPRE/SDE 2433/14

A business tax and an extraction tax will be applied by the local government on the project's revenue. Taxes are calculated based on the annual rental value of the fixed facilities required. For Achmmach, the rental value is determined as 3% of the value of the facility and is capped at MAD50 000 000 (approximately US$ 5.5M). The business tax rate for mining activities is set at 10% of the rental value, or MAD150 000pa (approximately US$ 16,500) for Achmmach. However, newly incorporated entities are exempt from the business tax for a period of five years from commencement of activity. The extraction tax is set at MAD3/t of the product leaving the mine gate. This amounts to MAD36 000 (approximately US$4,000) per annum for the Achmmach operations.

Annual rental costs on LE 332912, including land rental costs with local communes, equates to approximately MAD 365 000 per annum (approximately US$ 40,000)

12.2.3 Permits

Atlas Tin requires a number of permits in order to operate the project. These include permits to operate, for the construction of a water storage facility, explosives, powerline and the decision to construct. Atlas Tin continue to progress these permits. Table 12-2 below outlines the current state of land agreements and permits required by Atlas Tin.

Table 12-2 Current list of land agreements

Approval	Date issued	Expiry	Authority
LE 33 2912 (Mining Lease)	12/03/2010	17/01/2022 (subject to renewal)	MEM Fes
Environmental & social (ESIA and ESMMP)	22/12/2014	22/12/2019	Ministry of Environment. Decision No. 36/2014

Approval	Date issued	Expiry	Authority
Collective land rental agreement	25/11/2014	25/11/2026	Ministry of Interior. File 10776- Renewal application submitted on May 2017
Community development agreements	17/10/2014	By mutual consent	Presidents of Ait Ouikhalfen and Ras Ijerri communes
Project water supply and management	19/12/2014	Open	ABHS, Fes. ABHS/DEPRE/SDE 2433/14
Water extraction authorization	1/12/2012	31/12/2031	ABHS Fes Decision N° 1540/12
Development of support water bores	09/02/2017	19/02/2019	ABHS, Fes. Licence No. 243/2015
Permis d'Occuper	17/06/2015	17/06/2018	Decision No. 02/2015 Communes, WALI Meknès/Tafilalet
PO exploration	Under renewal process	TBA	Decision of Wali Fes-Meknes
Water Storage facility 400 000m^3	Need application, design & impact study	TBA	ABHS Fes
Power transmission	Memorandum of agreement to be signed with ONEE	TBA	ONEE and CRI
Decision to Construct	Not yet applied for	Life of mine	Communes and Provincial Governor
Explosives permit	Not yet applied for	Life of mine	Communes, Provincial Governor and MEMEE

12.2.4 Archaeological and Cultural

There are no archaeological or formal cultural sites in the project area. However, there are a few spiritual sites in the concession area and its vicinity, among which is the Sidi Addi peak, which has spiritual significance for the local people as it is considered to be the mausoleum of the Saint Addi. Atlas Tin's Achmmach project does not affect these sites.

12.3 Legal

12.3.1 Sovereign Risk

Morocco is a constitutional monarchy, led by King Mohamed VI who succeeded to the throne in 1999.

There are two legislative bodies, the Lower House, which is directly elected, and the upper house, elected through a college of electors. The last general election took place in October 2016, with the Islamist Justice and Development Party (PJD) gaining power. The next general election is due in 2021.

In 2011, as the 'Arab Spring' reverberated through much of the region, Morocco was relatively unaffected, experiencing only minor protests compared to other affected countries. As a result, Morocco adopted a new, more progressive constitution and elected a new Government the same year. Comprehensive reform agendas continue to be implemented.

Morocco was granted Advanced Status in 2008 by the EU, a reflection to the strong economic and political ties it holds with Europe. Morocco also enjoys strong relations with Sub-Saharan Africa, which are increasingly being strengthened by frequent exchanges of official visits and economic partnerships. Morocco officially re-joined the African Union in January 2017 and has applied to join the Economic Community of West Africa States (ECOWAS).

Territorial borders with neighbouring Algeria have been closed since 1994 with both countries continuing to have diverging views on several issues, mainly the undetermined status of Western Sahara, currently under de facto Moroccan administration.

12.3.2 Economic Development

Not only is Morocco an important market in its own right, it is also becoming the premier commercial gateway to Africa, as well as Africa's bridge to Europe. Africa presents the single largest growth opportunity in the world and is a new frontier for many international businesses.

Morocco, with its strategic geographic position and its intimate knowledge of the continent, will increasingly become a major gateway to these markets.

Over the last decade, and with particular focus in the past few years, Morocco has reformed its economy to enhance productivity and strengthen resilience to external shocks. Key reforms have included restoring macro-economic balance and cutting subsidies. Growth has averaged 4.5% with agriculture, phosphate exports, and remittances as the main contributors to GDP.

Diversifying the economy is a government priority as set out in the 2014-2020 Industrial Acceleration Strategy which includes key industries, including mining, renewable energy, automotive, aerospace, cabling, textiles and pharmaceuticals. If successful, the strategy is expected to generate half a million jobs and increase the share of industry in GDP by 9%. Following agriculture, mining activities contributed the most to growth, mostly driven by the mining of phosphates. Inflation remains low at 0.7 percent. Unemployment remains on an upward trend, rising from 9.9 in 2016 to 10.2 percent in 2017.

Morocco improved its score in the World Bank's 2017 Ease of Doing Business index, ranking 68 over 190 countries, and moving up 19 places since 2014. A 2013 Financial Law made the improvement of the business environment a priority to boost foreign investment.

12.3.3 Legal System

Morocco has a constitutional, democratic parliamentary and social monarchy. Moroccan law is shaped by French Civil Law and a combination of Muslim and Jewish traditions. The Constitution of Morocco has also played a pivotal role in shaping the law and legal systems in Morocco. The most recent constitutional development was the adoption of a new Constitution in July 2011.

Morocco has a dual legal system consisting of secular courts based on French legal tradition, and courts based on Jewish and Islamic traditions. The secular system includes communal and district courts, courts of first instance, appellate courts, and a Supreme Court. The Supreme Court is divided into five chambers:

- criminal
- correctional (civil) appeals
- social
- administrative
- constitutional.

The Special Court of Justice may try officials on charges raised by a two-thirds majority of the full Majlis. There is also a military court for cases involving military personnel and occasionally matters pertaining to state security. The Supreme Council of the Judiciary regulates the judiciary and is presided over by the king. Judges are appointed on the advice of the council. Judges in the secular system are university-trained lawyers.

12.3.4 Territorial Organisation and Land Tenure System

The territorial organisation and land tenure system in Morocco is characterized by a plurality of legal status. Most of the land in the project area are collective land. This is administered by a Jmaâ (assembly) composed of Naibs (representatives of the land right-holders), under the direct supervision of the Ministry of Interior (through the Directorate of Rural Affairs-Direction des affaires rurales).

Land covered by the Operating Licence falls into three categories:

- Collective plots belonging to an ethnic community. These are non-transferable but can be allocated, (ie shared in lots distributed to beneficiaries), or not allocated (ie exploited for the benefit of the whole community). Within the boundaries of operating licence LE 332912, most plots are collective plots allocated to beneficiaries and used for pastoral purposes. They are managed by a committee (Jmaâ) made up of beneficiary representatives (Naibs), under the administrative supervision of the Ministry of Home Affairs (via the Rural Affairs Department). All decisions concerning these plots must be approved by the Naib committee. The Jmaâ steers the management of collective plots, from division rules to fragmented plot organisation. In most cases, farmland is rented by the beneficiaries to farmers who pay up to a third of their harvest or 1000 dirhams per hectare per year as rent.
- Melk plots, whatever their origin (purchased, inherited, donated, etc.) belong to one or several people with full ownership and enjoyment rights: they are private property.
- The forests are the property of the State: they are managed by the provincial delegation of the High-Commission for Water & Forests and for Desertification Prevention (HCEFLCD).

12.4 Site Rehabilitation Obligations

The Mining Law requires titleholders to prepare an abandonment plan, the contents of which are to be set out in subsequent regulations. To date, no regulations have been published in the Official Journal to confirm the contents of the abandonment plans.

However, article 6(4) of law no. 12-03 on ESIAs provides that the abandonment plan must include the "measures envisaged by the applicant to remove, reduce or off-set the negative consequences of the project on the environment, as well as measures to enhance the positive impacts of the project". As the ESIA is a prerequisite to the approval of the project and the applicant undertakes to implement the provisions of the ESIA, the approved rehabilitation measures under the ESIA will form part of the requirements for the execution of the project.

Atlas Tin as part of the ESIA and the design of the project has outlined in accordance with the IFC Standards and Equator principles closure planning and included closure costs in the economic modelling for the project. These are detailed in Section 9.3 of this feasibility report.

12.5 Tax

The Moroccan taxation system consists of direct and indirect taxes. Indirect taxes provide a greater source of tax revenue than the direct taxes. Moroccan corporations are subject to a unitary tax system called the corporate tax (impôt sur les sociétés or IS). The Moroccan tax system has been codified under the General Tax Code.

Investors are also entitled to certain incentives under the general investment laws in Morocco. These incentives are set out an investment charter, together with the General Tax Code. The Investment Charter also provides that companies intending to meet stipulated minimum thresholds may enter into a specific contract (Investment Agreement) with the State setting out additional advantages beyond those included in the Investment Charter. The Achmmach Tin Project meets the stipulated minimum threshold and consequently Atlas Tin intends to enter into an Investment Agreement with the State. The Investment Agreement will provide long term fiscal stability for the Achmmach Tin Project. For the purposes of the DFS, Atlas Tin has only allowed for concessions available in the Investment Charter.

12.5.1 Corporate Income Tax

In Morocco, the corporate income tax is based on the net income companies obtain while exercising their business activity, normally during one business year. Corporate tax is payable in four equal instalments based on the prior year's assessment. The corporate income tax rate has been stable since 2008 at 30%, before being revising upwards to 31% in 2016. Corporates which export their products benefit from a reduced corporate income tax rate of 17.5%. Companies are always subjected to the higher of a legal minimum tax (cotisation minimale (CM)) of MAD 1,500 or 0.5 % of the annual turnover. Brought forward operating tax losses can be carried forward for a maximum period of four years whilst depreciation losses can be carried forward indefinitely.

Atlas Tin intends to export all of the concentrate produced from Achmmach and consequently will benefit from the reduced tax rate of 17.5%.

The Investment Agreement provides for an investment incentive regime whereby the State may provide financial support of up to 5% of the overall costs of the investment. The substance of these incentives will need to be negotiated with the State. The incentive framework provides for:

- Up to 20% of the costs relating to the acquisition of the land acquired for the purposes of the investment. Atlas Tin is unlikely to utilise this benefit as land acquisition is not required for the Achmmach Tin Project.
- Up to 5% of the costs of external infrastructure. Atlas Tin has assumed that it will benefit from this benefit for roads and other external infrastructure to be constructed as part of the Achmmach Tin Project.
- Up to 20% of costs in respect of professional training. No professional training incentive has been recognised in the project financial model. However, additional training or upskilling be required, this incentive is available to be accessed by the company.

The incentives available will need to be formalised in an Investment Agreement. However, the framework has successfully benefit other significant projects and demonstrates the appetite for large investments of the Moroccan government.

The Moroccan General Tax Code provides for an accelerated depreciation regime. However, companies who benefit from, concessional income tax rates, would not be able to claim the accelerated depreciation benefit. Atlas Tin will therefore adopt the preferred straight line depreciation methodology based on the suggested depreciation rates stipulated in the General Tax Code. Depreciation tax losses can be carried forward indefinitely, unlike operating losses which expire within 4 years.

12.5.2 Value Added Tax

Value Added Tax (VAT) is levied under the Moroccan Tax Code and is due on all industrial, commercial, and handicraft transactions taking place in Morocco, as well as on importation operations. The standard rate of VAT is 20%. Lower rates of 7%, 10%, and 14% apply to specifically designated operations.

The General Tax Code allows a 36-month VAT exemption for equipment purchases for projects exceeding MAD 100 million (approximately US$11 million), commencing from the date of delivery of construction authorisation.

The sale of goods is considered as taking place in Morocco, and thus subject to VAT, if the goods sold are delivered in Morocco. The sale of services is considered as taking place in Morocco, and thus subject to VAT, if the services sold are consumed or used in Morocco.

VAT refund claims are lodged on a quarterly basis. Refunds are paid by the authority within 3 months following the claim. An allowance of 6 months for refunds has been included in DFS.

As the investment in development of the Achmmach Tin Project exceeds the criteria for the 36-month VAT exemption, Atlas Tin has assumed that this 36-month exemption will be applicable from construction commencement.

12.5.3 Withholding Taxes

Withholding taxes of 10% are applicable for payments to non-resident service providers, subject to the provisions of double tax treaties between Morocco and the country of the residence of the foreign service provider.

Withholding taxes of 15% (unless reduced by treaty) are applicable to dividends declared to entities not subjected to Moroccan corporate income tax (non-resident shareholders).

Withholding taxes of 10% (unless reduced by treaty) are applicable on any dividend paid to foreign lenders. Interest paid to the foreign lenders are exempt from withholding taxes if the loan is in foreign currencies and has a tenor of at least 10 years.

12.5.4 Local Business Tax

Companies are subjected to annual local business taxes, communal services tax and tax on mining operations (extraction tax).

Local business tax is computed by the tax administration at the rate of 10% of the rental value of the assets owned or leased by the company, capped at MAD 50 million (approximately US$5.6 million) per annum. Companies will benefit from local business tax exemption of 5 years for each new asset purchased.

Communal services tax is due at the rate of 6.5% on the assets owned. These local business taxes are deductible in the computation of corporate income tax.

12.5.5 Extraction Tax

Extraction tax tariff ranges from MAD 1 to MAD 3 per extracted tonne. The amount of tariff levied on Atlas Tin is subject to negotiation with local authorities and will be formalised in an Investment Agreement. Atlas Tin has assumed an extraction tax tariff of MAD 3 per extracted tonne in the DFS.

Extraction taxes/royalties in Morocco are payable to the National Office of Hydrocarbons and Mines (ONHYM), a financially autonomous public body. ONHYM is responsible for initial exploration before partnering with the private sector to exploit resources.

Atlas Tin currently is subjected to a MAD 100,000 (US$ 11,000) per annum royalty until production commences. Once production commences, royalty is notionally payable at 3% of net smelter royalty. Atlas Tin has allowed for this maximum royalty in the financial model. However, the company will seek to negotiate the terms of the royalty arrangement with ONHYM with a view towards reducing this amount.

12.6 Contractual Arrangements

12.6.1 Construction

The EPC contract for the process plant design and construction represents the major expenditure of the project execution phase. Tenderers for the works will be established through an expression of interest and shortlisting process. It is anticipated that the form of contract will be the FIDIC Silver Book, suitably modified in conjunction with Atlas Tin's legal advisers. FIDIC forms of contracts are widely used in Europe and Western Africa. The contract will allow for the provision of guaranteed performance (time, cost, and physical process capacities) by the EPC Contractor.

12.6.2 Mining

Atlas Tin will undertake a tender for mining contractors from Australia, Europe or Africa. It is intended that the mining contract will be based on a schedule of rates with the appropriate mechanism to ensure delivery against the mine plan.

12.6.3 Operations

Atlas Tin will develop standard contracting terms and conditions in collaboration with its legal advisers. These standard terms and conditions will broadly cover the following areas:

Long term supply arrangements

Atlas Tin will enter into contractual agreements with standard terms of business with suppliers who have been awarded the contracts for long term supply of goods, consumables and services. The terms and conditions will include performance measures, incentive and penalty arrangements, and clearly define the obligations of the supplier to minimise the risk and exposure to Atlas Tin.

Purchases

One-off purchases will be procured under standard Atlas Tin purchase orders. Purchase orders will be issued subject to a delegated authority framework.

Labour agreements

Atlas Tin will develop two standard form labour agreements: expatriate employment contracts; and a local workforce employment contract.

12.6.4 Marketing

The Achmmach Tin Project will produce high quality tin concentrate. Atlas Tin has not entered into any agreements or heads of agreement with respect to the sale of concentrate.

The company intends to market its product to a third party offtaker or offtakers, which will be selected through a competitive tender process. Pursuant to the terms of the Shareholder Agreement, each of TTC and NMC has the right to market its proportion of tin concentrate, subject to meeting commercially competitive terms.

The dynamics and forecast demand for tin in the foreseeable future is discussed in Section 13. Based on the market dynamics and informal expression of interest, there is a high level of confidence that the Achmmach Tin Project Concentrate will be marketed at commercially competitive pricing.

For the purpose of this definitive feasibility study, Atlas Tin has obtained indicative smelter terms from a number of leading international tin smelting and trading groups to determine the commercial terms modelled in the study.

13 MARKET ANALYSIS

13.1 Product Specification

Achmmach mine will produce a clean 60% tin concentrate. At peak production the mine will produce 8,000 tonnes of concentrate per annum.

13.2 Tin Market Analysis

This analysis of the tin market is based on several tin commodity reports and includes material sourced from the International Tin Association (ITA).

13.2.1 Tin Usage

The tin market saw a contraction in demand during 2015 of 3.8%. In 2016 demand grew by 2.5% and is estimated to have grown by 3.2% in 2017 to 357,000 tonnes of consumed refined tin, with all major global economies contributing to the growth.

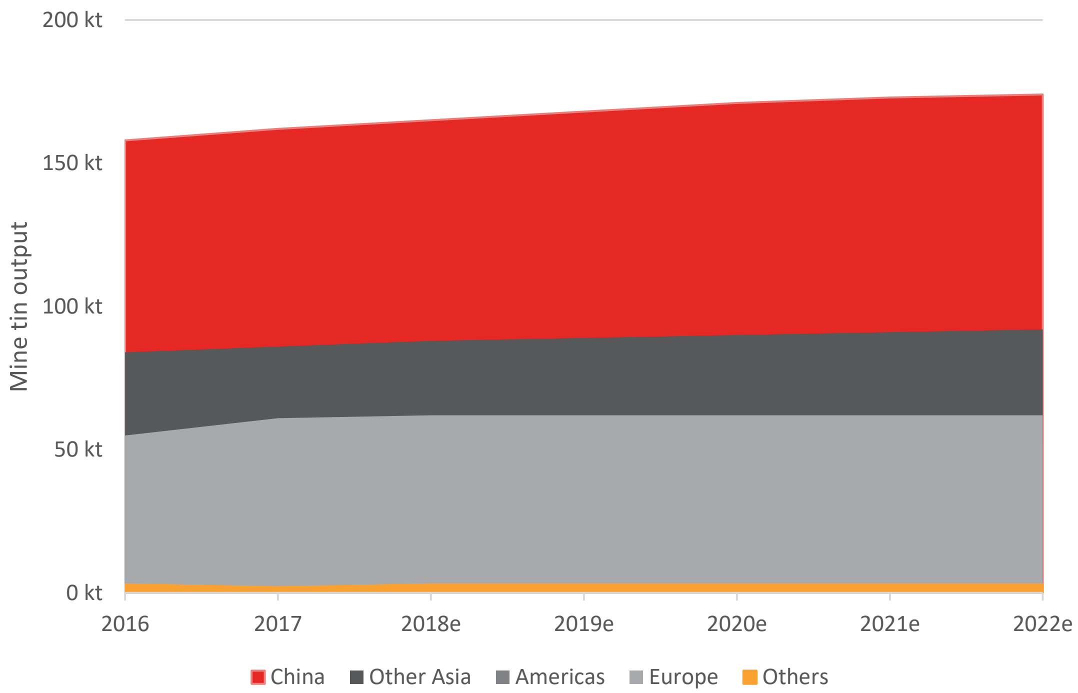


Figure 13-1 Refined tin consumption (source: ITA)

The World Bank and IMF are forecasting global growth of between 3% and 4% in 2018 and 2019 respectively. A conservative view for tin demand through to 2030 is steady growth of 1.1% per annum.

According to ITA, the most likely scenario for refined tin use over the next 10 years is illustrated in the chart below, which reflects an impact on consumption from technological developments and substitution (i.e. improvements in efficiencies in the use of tin), offset by non-technological drivers such as global economic growth and tin price trends. Optimistic and pessimistic scenarios for tin demand have been modelled together with an intermediate (forecast) scenario under which the global tin demand is expected to grow by around 1.1%pa to 2030.

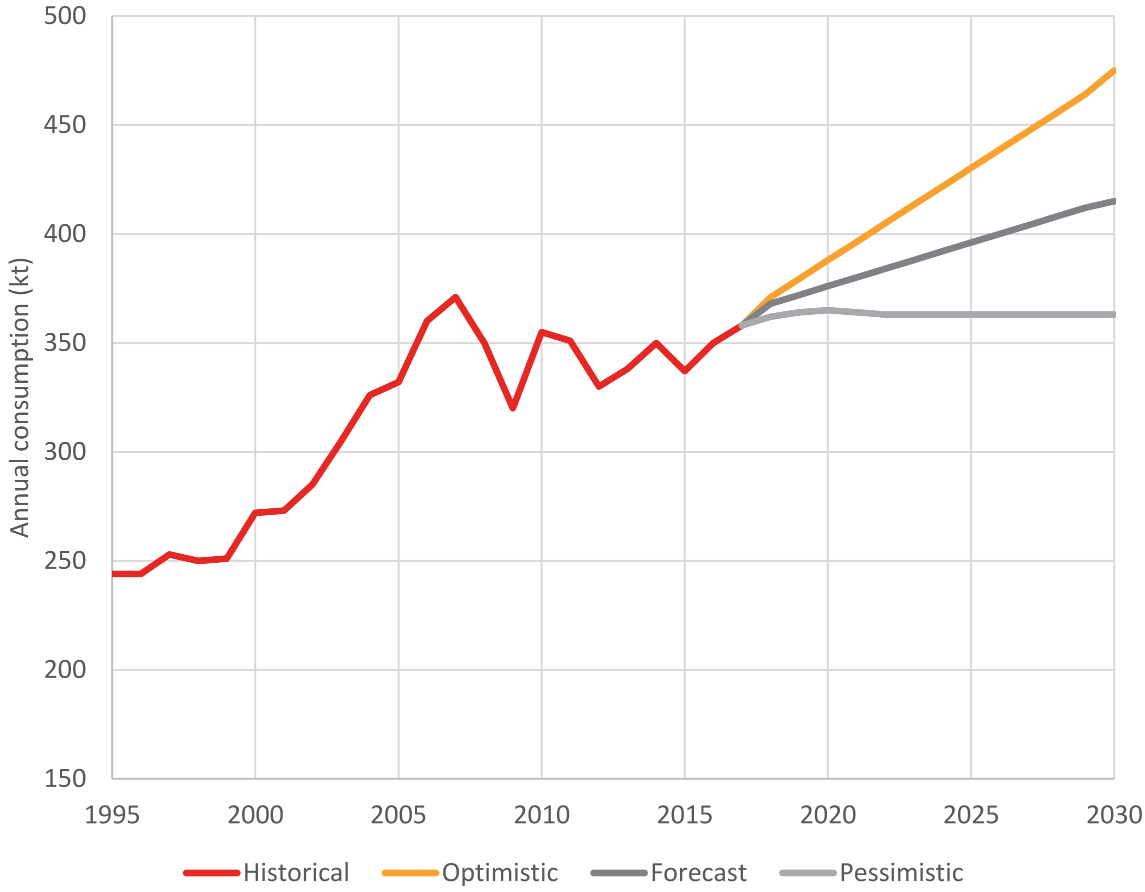


Figure 13-2 Refined tin use projections to 20130 (source: ITA)

Assessments of short and medium-term technology threats and opportunities show that threats to solders, notably from electronics miniaturisation, and in the tin plate sector, may be partially balanced by opportunities in energy related technologies, notably lead-acid batteries and other energy materials.

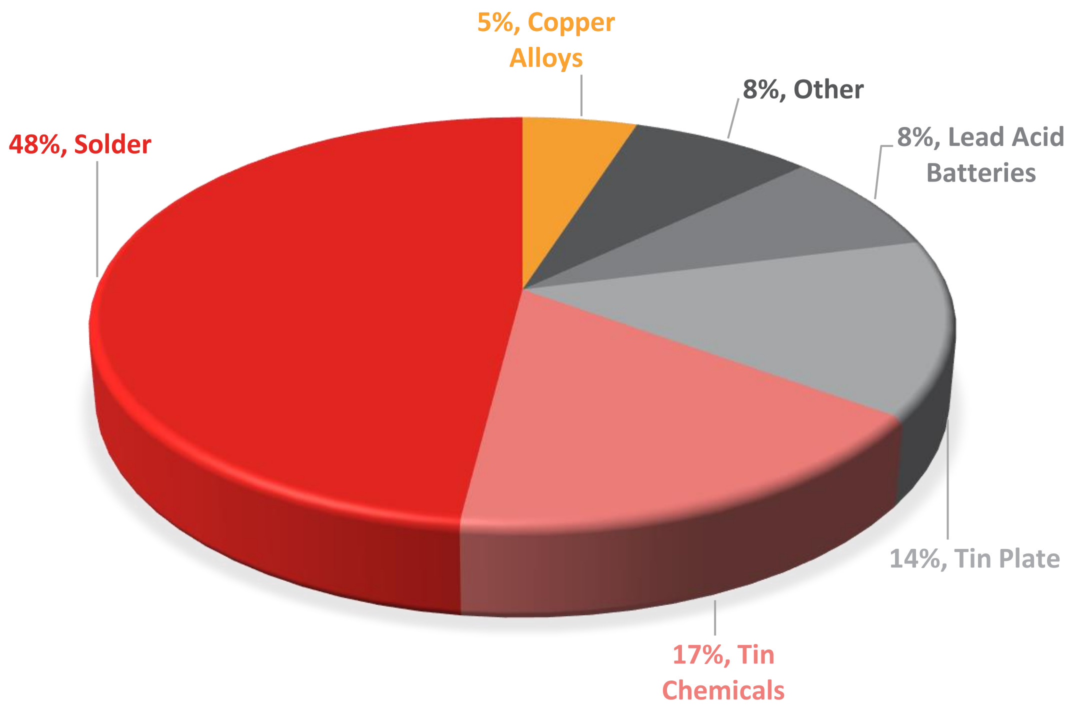


Figure 13-3 Tin usage by sector

13.2.2 Solder

Solder, representing broadly half of global consumption, is the largest consumer of tin as tin is the primary component of both leaded and lead-free varieties used in electronics. There has been a growing trend in recent years to move towards lead-free solder, where tin forms approximately 97% of the product. The long-term outlook for solder usage remains balanced, with growth in electronics and further conversion to lead-free solders in high reliability applications, such as aerospace and military, offset by smaller unit volumes as a result of miniaturisation

13.2.3 Chemicals

Tin use in chemicals is the second largest tin application and looks likely to retain this position for the foreseeable future. Important tin chemical applications include PVC stabilisers, polyurethane foam manufacture and glass coatings.

13.2.4 Tin Plate

Tinplate, primarily used in food and beverage cans, is a very traditional market for tin which functions as a corrosion protector in the material. Until recently, tin use in the sector has remained largely stable, with little change over the last decade. Continued competition from aluminium beverage cans and other alternative competitive packaging materials are making inroads into this traditional market. In the United States, tin coating weights increased due to the introduction of BPA-free lacquers in food cans.

13.2.5 Lead Acid Batteries

Tin is added to lead acid battery grids to improve casting and performance. It is also used in posts and straps connecting the grids as well as in solder joining components. Lead acid batteries primary use is in internal combustion engines (cars, trucks, boats, ships etc.), where the battery is used to turn the engine over at ignition, and then to operate the various electrical components in the vehicle – air conditioning, electric windows, wind-screen wipers etc. The tin content in lead acid batteries has been increasing and the demand for tin in lead acid batteries is predicted to grow at 2-4% per annum from 28,200 tonnes in 2016 to 36,500 tonnes by 2025, after which there is a risk of substitution by lithium-ion and other technologies. Lead acid batteries remain the lowest cost and most widely used solution for energy storage in rechargeable batteries, a sector expected to grow with the increased reliance in renewable energy source.

13.2.6 Copper Alloys

Tin and copper are traditionally combined to produce bronze, but tin is also added to other copper based alloys such as brass.

13.3 Tin Supply

Maintaining world tin production over the next five to ten years will continue to require a strong market environment and the development of new mines. Declining or static mine production from existing operations in Indonesia, Myanmar, Peru and China are forecast in the medium term. Existing operations in Peru and Indonesia face pressure from depleting resources and falling grades. Chinese mine output has remained quite stable and this is expected to continue in the medium-term, although output may recover to partially offset an anticipated decline in tin shipments from Myanmar due to rising costs and depleting resources. Tin production from inner Mongolia is expected to grow in 2018 and beyond as a result of tin discoveries and investment in operations there.

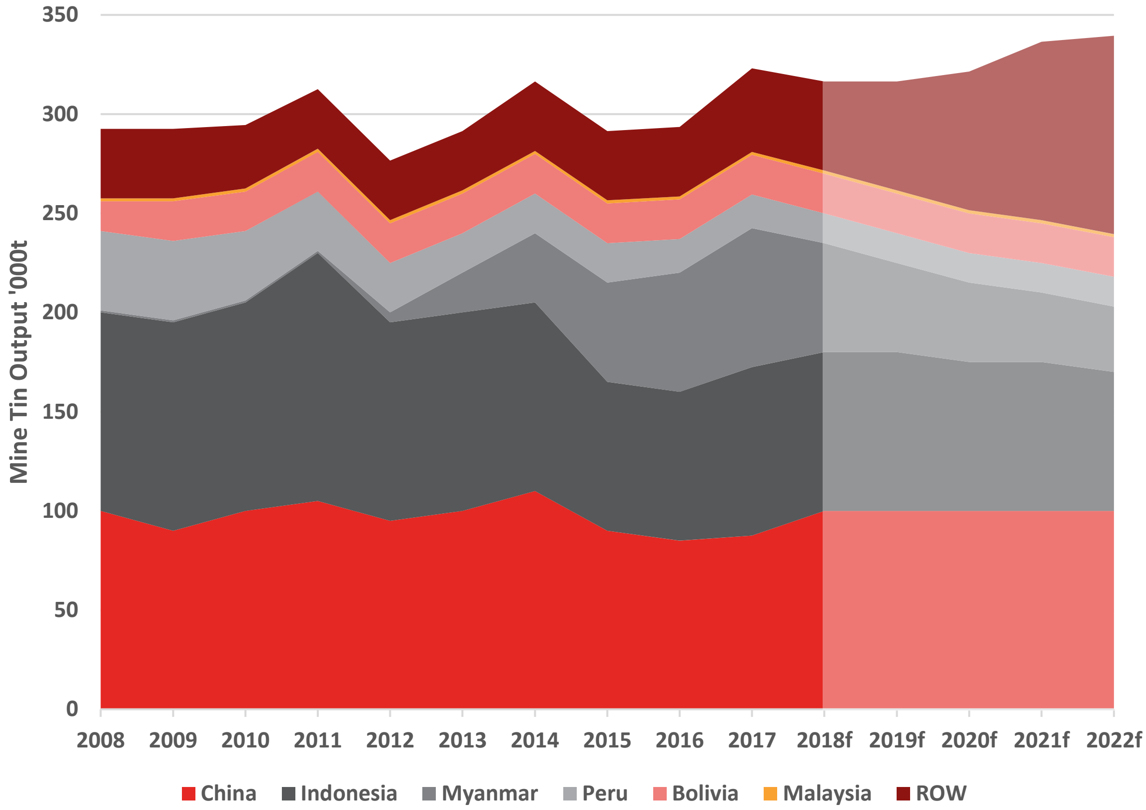


Figure 13-4 Tin mine production by country

Higher tin prices will be required to justify the development of new tin mines and or incentivise additional small-scale and artisanal mine production to replace declining production to offset rising demand. It is forecast that by 2022 approximately 65,000 tpa (tonnes per annum) of additional capacity will be required. Without new supply, mine production rates could fall to 270,000 tpa. Secondary production is expected to remain relatively stable at approximately 60,000tpy although higher prices could boost recycling.

The recent recovery in tin prices has led to a resurgence of tin development activity, with 21 tin projects identified by the ITA (Table 13-1 below) which could potentially achieve 72,700 tpa of production by 2022 if the projects can be demonstrated to be feasible, funding is secured and they are built.

Table 13-1 Identified tin projects

Country	Company	Mine/Project	Design capacity (tpa)	Start up year
Russia	Khingan Resources	Khinganolovo tailings	800	2018
Namibia	Afitin	Uis	800	2018
China	Weilasituo Mining	Weilasituo	6,000	2018
Australia	Elementos	Cleveland	2,000	2019
China	Yunnan Tin	Yunnan Tin Wuchangping	2,000	2019
Australia	AusTin	Taonga	2,800	2019
Peru	Minsur	San Rafael Tailings	4,750	2019

Nigeria	PT Timah	PT Timah Nigeria JV	5,000	2019
DR Congo	Alphamin Resources	Bisie	10,750	2019
Canada	Avalon Advanced Materials	East Kemptville	1,500	2020
Australia	Stellar Resources	Heemskirk	4,400	2020
Morocco	Kasbah Resources	Achmmach	4,500	2020
Australia	Bluestone Mines Tasmaina JV	Rentails	5,400	2020
Kazakhstan	Tin One Mining	Syrymbet	9,100	2021
Brazil	Brazil Tin	Arara	500	2022
Germany	Tin Interational	Sadisdorf	700	2022
Czech Republic	European Metals	Cinovec	1,000	2022
Australia	Avira Resources	Mount Veteran	1,000	2022
Australia	Australian Tin Resources	Ardlethan Tailings	1,700	2022
Australia	Ventrure Minerals	Mount Lindsay	3,000	2022
United Kingdom	Stronbow Exploration	South Crofty	5,000	2022

This data is represented graphically in Figure 13-5 below. For clarity, the Cinovec resource is not shown as it is a tin/lithium deposit with an equivalent grade (and very low tin grade) which is difficult to reconcile with this plot.

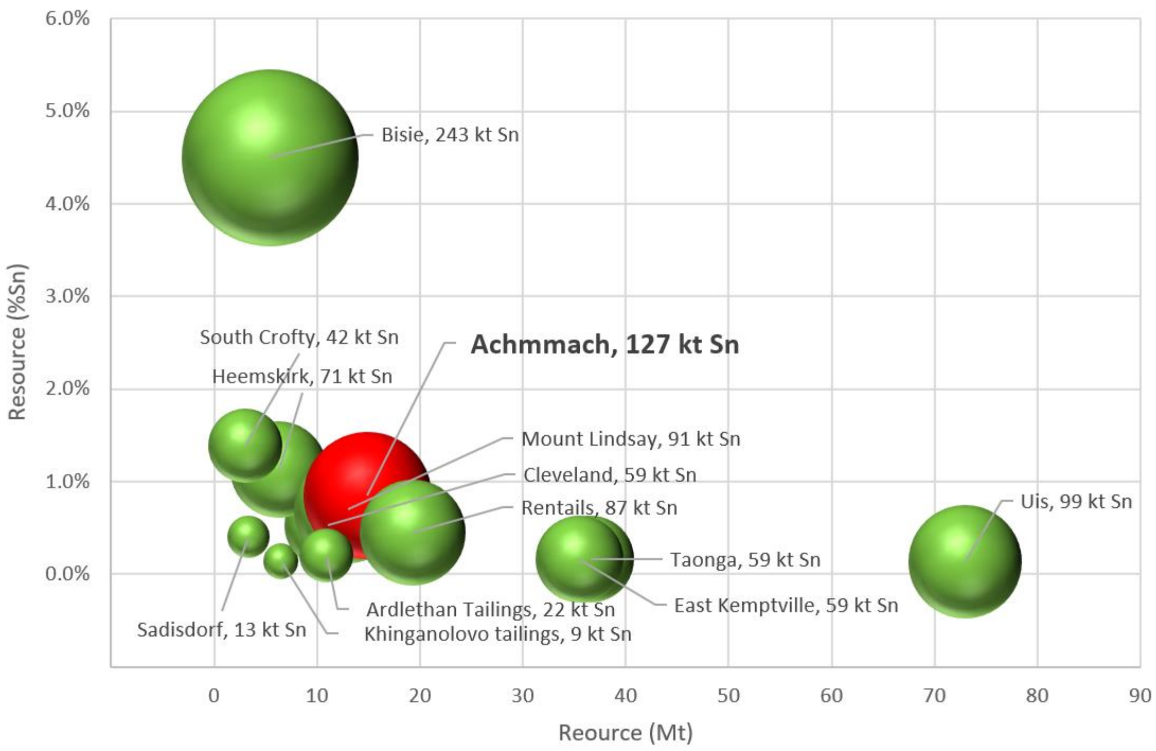


Figure 13-5 Comparison of developing projects (bubble size indicates contained metal)

It is optimistic to assume that all 21 projects will prove feasible and the likely outcome is that only a select number of these projects will actually be developed in the short to medium-term. Even these projects are likely to be subject to delays in commissioning and ramping up to design capacity. In fact, with a search through company websites, 10 of the listed projects do not currently have JORC reserves (only resources), representing 26,300t of potential tin production capacity. (Seven of the companies listed do not have easily accessible websites). Four projects with published ore reserves are listed in Table 13-2.

Table 13-2 Projects with public ore reserves

Mine/Project	Resource (Mt)	Resource grade (%Sn)	Reserve (Mt)	Reserve grade (%Sn)	Design capacity (tpa)	Start up year
Khinganolovo tailings	6.5	0.14%	6.0	0.14%	800	2018
Cleveland	11.1	0.53%	3.7	0.29%	2,000	2019
Bisie	5.4	4.50%	3.5	4.34%	10,750	2019
Achmmach	14.9	0.85%	7.0	0.82%	4,500	2020

This suggests that the project development list in Table 13-1 represents an upside view and that the required 65,000 t of new tin supply required by 2022 will not be achieved.

13.4 Market Balance and Outlook

Global stocks of tin remain at historically low levels, shown below in Figure 13-6.

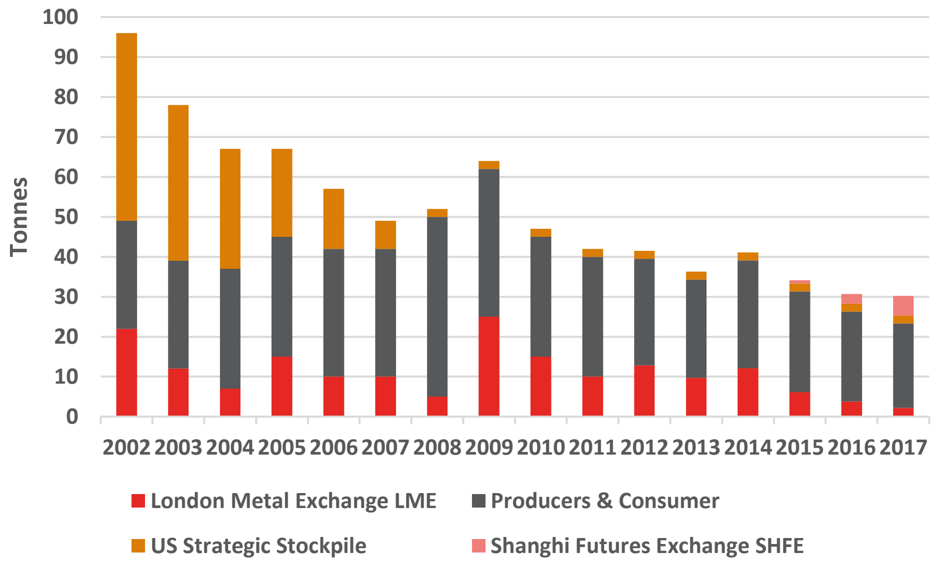


Figure 13-6 Historical tin stocks

Stocks of 25,000 tonnes of tin currently represent just 36 days of global consumption, and as such will not act as a buffer to price changes.

The forecast market balance for tin is shown in Figure 13-7 below.

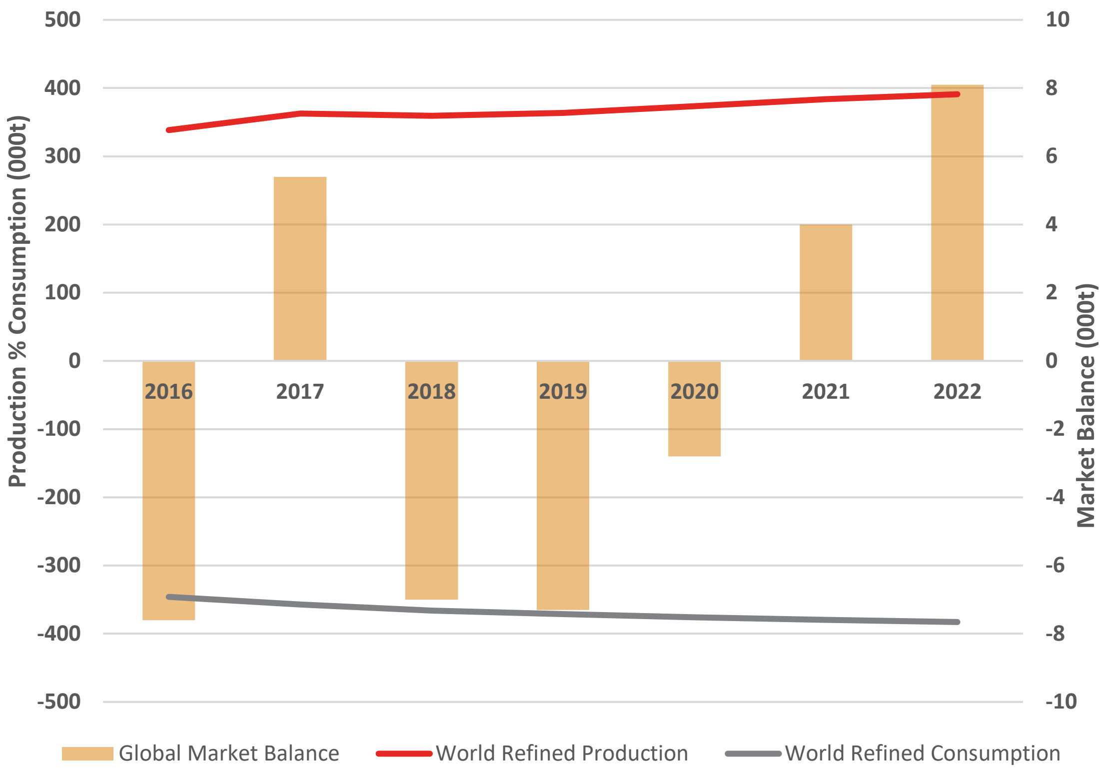


Figure 13-7 Forecast market balance for tin (source: ITA)

The market has been in deficit in recent years and is expected to remain so in the near term. The surpluses shown for 2021 and 2022 assume that all 21 projects identified by the ITA come into production by 2022, which is regarded as unlikely. Market deficits are likely to continue in the future and as a result prices are forecast to increase to correct the market imbalance.

As a result of current and forecast stock levels and market balance three forecast price scenarios are provided (Figure 13-8).

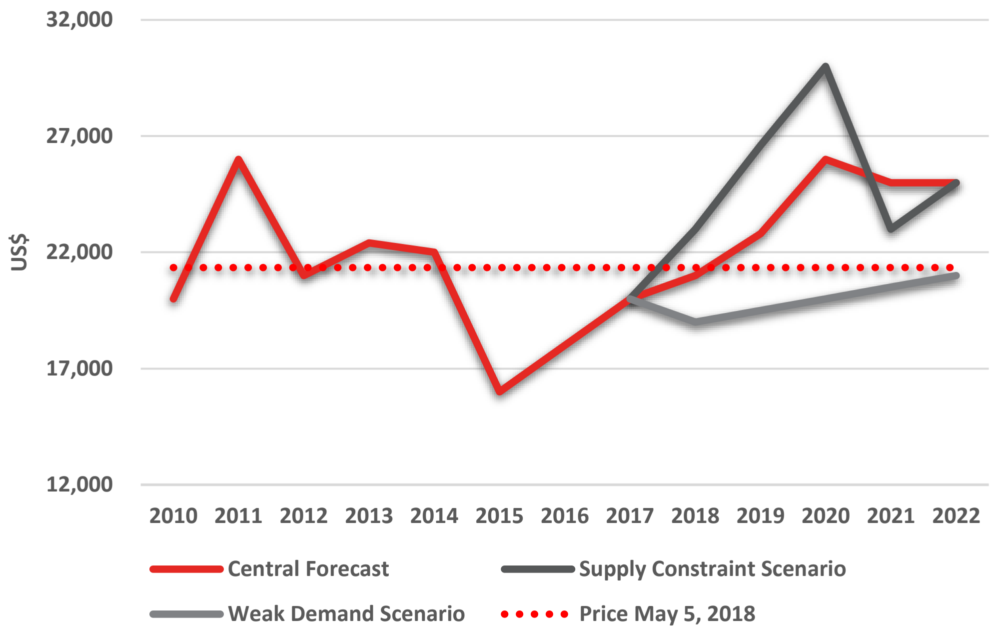


Figure 13-8 Forecast prices for tin (Source: ITA)

The mid-term equilibrium price required to balance supply and demand is around US$25,000 in 2022, linked primarily to increases in mine production costs, and prices are expected to trend towards this level over the next four years.

The supply constraint scenario reflects a situation where tin supply is less than anticipated (e.g. as a result of projects not commencing or commencing later than currently forecast) and can't offset declining production quickly enough to meet rising demand.

The weak demand scenario anticipates weaker global tin consumption as a result of a weaker economic growth. Alternatively, demand could be impacted by a bigger impact on demand from technological innovation, accelerating economisation of tin use and or development of material substitutes.

The base tin price used for financial modelling purposes in this feasibility study is US$21,000.

14 CAPITAL COST ESTIMATE

As part of the process of developing the Project into a functioning mine, Lycopodium ADP was engaged by Atlas Tin to develop the capital cost estimate (capex) for the processing plant and key site infrastructure.

Entech Mining Consultants was engaged by Atlas to develop detailed mine physical schedules. These schedules were provided to mining contractors to develop mine capex on a schedule of rates basis.

Certain capex estimates have been sourced from local Moroccan and international suppliers.

These capex estimates were based on 2018 prices.

14.1 Basis of Estimate – Capital Cost

14.1.1 Estimate Accuracy

For the scope of work covered and the estimating methodology applied, capital costs estimates sourced by Lycopodium ADP and from third party supplier quotes have an average contingency of 12% applied.

The CAPEX estimate has an accuracy provision of +/-12%.

14.1.2 Estimate Base Date

The estimate base date for the Definitive Feasibility study is 1 April 2019.

14.1.3 Exchange Rates

The project capex estimate is reported in United States dollars (US$), with all other currencies converted to US$ at the exchange rates listed in Table 14-1.

Table 14-1 Exchange rates

Currency	Rate of exchange to USD
AUD	0.745
ZAR	0.075
EUR	1.157
MAD	0.105

14.1.4 Estimate Format and Build-up

The estimate has been estimated based on the following sources of data:

- Lycopodium ADP's in-house estimating systems
- mining contractor submissions
- third party quotations
- first principles cost development.

14.2 Definition of Costs

Definitions used in the capital cost estimate and the associated basis are summarised below and detailed more fully in the capex estimate.

14.2.1 Mining Development

Mining development includes the cost incurred for the initial development of the underground mine including, but not limited to, mining contractor mobilisation, lateral and vertical development, haulage, fixed power, labour and day works and mine infrastructure costs such as ventilation, safety, electrical and communications fixed plant, etc.

14.2.2 TSF and WSF

TSF and WSF includes earthworks associated with the initial cost during the pre-production construction phase. Subsequent TSF raises have been treated as sustaining capital.

14.2.3 Process Plant

Process plant includes the cost incurred for the construction of the process plant including, but not limited to, crushing, grinding, ore sorting, tin concentration, civil and earthworks, structural supply and installation, plate work supply and installation, mechanical supply and installation, electrical and instrumentation (E&I) supply and installation, piping and transport The process plant capex also includes the earthworks associated with the ROM pad.

14.2.4 Engineering, Procurement and Construction (EPC)

EPC costs is the costs of a reputable engineering firm to manage the engineering and procurement and deliver the construction of the process plant on a fixed costs basis.

14.2.5 Infrastructure Costs

Infrastructure costs includes buildings, bulk earthworks associated with administrative buildings, roads, power line, water supply and distribution, sewage systems, etc.

14.2.6 Indirect Costs for Construction

Indirect costs are those costs incurred in the development of the mine that are not directly assignable to “Direct Costs”. Indirect costs include, but are not limited to owner’s vehicles, vendor indirect, community resettlement, IT software, fit out and equipment, consultancy allowances, construction insurance, etc.

14.2.7 First Fills and Spares

First fills, initial capital inventory items, including reagents, lubricants and grinding media. Spares include strategic mechanical and E&I spares.

14.2.8 Financing Costs

Financing costs are those financial charges incurred as a result of raising debt and capital over the course of the project. The study has been presented on an ungeared basis and therefore excludes financing costs and any related tax shields resulting from finance cost tax deductions.

14.2.9 Pre-production costs

For the purpose of this estimate, pre-production costs are defined as those costs incurred from the commencement of construction to the start of project ramp-up.

14.2.10 Sustaining Capital

Sustaining capital is defined as the capital that is required to maintain production at design levels and includes the underground development required once production has commenced, on-going mine infrastructure, TSF raises and replacement of other equipment throughout the life of mine.

14.2.11 Estimate Strategy and Presentation of Estimate

The estimating strategy comprises a defined level of work and/or engineering deliverables required to support the DFS study and the application of a defined estimating methodology to deliver a capex estimate, with the required estimate accuracy provision. The estimating methodology, including level of work undertaken in support of the DFS and the estimating basis is defined below.

The purpose of the capital cost estimate is to establish the project budget to sufficient accuracy to allow investment decisions. The estimate provides the pertinent cost data to establish the control budget for moving into project execution (procurement and field construction). The 2018 project capital costs are summarised in Table 14-2 below.

Table 14-2 2018 Project capital costs

Capital cost	US$ Millions
Mine development	12.1
TSF and WSF	3.5
Process plant	44.5
Infrastructure	12.0
Engineering Procurement and Construction (EPC)	7.2
Construction indirect costs	5.4
Sub Total – Project Construction Capital	**84.7**
First fills and spares	1.2
Contingency	10.5
Total Project Capital Costs	**96.4**

The capital cost estimate was compiled based on the following parameters:

- Foreign currency elements of quoted prices were converted to US$.
- An average contingency of 12% has been included based on the risk analysis.
- The quantum of contingency allowed is US$ 10.5 million.
- A combination of contractor submitted prices for mining contractors, developed from a physical schedule, and budget prices for the remainder of the minor mobile and fixed plant infrastructure and equipment obtained from a number of proven suppliers and estimates.
- Explosives quantities were estimated from first principles and applied to Moroccan explosives supplier price schedules.
- Firm proposals were received for the key equipment packages.
- Labour rates were based on rates validated by an independently sourced Moroccan mining industry human resources consultant.
- Expatriate rates include all required in-country taxes.

- Explosives quantities were estimated from first principles and applied to Moroccan explosives supplier price schedules.
- The costs for transport and freight were developed based on quotations from Moroccan service providers.

Detailed estimates of indirect costs for the project included:

- contractor general expenses
- bulk diesel for mobile vehicles
- road construction
- freight costs
- crew transportation
- general site indirect costs
- camp catering and cleaning
- pre-operational readiness and commissioning services
- operating spares, strategic and commissioning spares
- contractor assistance during commissioning.

Owner's costs, including:

- owners project management team
- owners pre-production site operations labour
- owners general & administration items.

14.2.12 Capital Cost Estimate Exclusions

The following costs are not included in the capital cost estimate:

- reimbursable taxes and duties
- schedule acceleration costs
- schedule delays and associated costs, such as those caused by:
    - unexpected site conditions
    - unidentified ground conditions
    - labour disputes
    - force majeure
    - permit applications
- development fees and approval costs beyond those specifically identified
- event risk
- financing costs other than what is included within the owners cost
- foreign exchange fluctuations (majority of quotes are US$ rate based)
- cost associated with third party delays
- development fees and approval costs of statutory authorities
- change in law and regulations
- no escalation within the capex estimate.

14.3 Capital and Pre-Production Costs Summary

14.3.1 Processing Plant

Table 14-3 below breakdowns the processing plant capital as developed for the 2018 definitive feasibility study.

Table 14-3 Processing plant capital

Processing plant capital	US$ Millions
Earthworks	4.7
Civil	3.6
Structural supply	2.9
Structural installation	1.0
Platework supply	1.5
Platework installation	0.3
Mechanical supply	17.2
Mechanical installation	1.1
Piping	2.1
E&I supply	5.8
E&I installation	2.9
Software	0.2
Transport	1.2
Total processing plant capital	**44.5**

Mechanical supply is summarised in Table 14-4 below.

Table 14-4 Mechanical supply

Mechanical supply	US$ Millions
Steinert Ore Sorter	2.1
Sandvik Crusher	2.1
NCP - Ball	1.9
Multotec Spirals	1.2
NCP - Regrind	1.2
Maelgwyn - Float Cells	1.1
Thyssen Krupp	0.9
Concentrators - Sepro System	0.8
Outotec - Sampler	0.8
Weir Slurry	0.6
Thickener	0.5
Conveyor equipment	0.5

Mechanical supply	US$ Millions
MT-Tables	0.4
Multotec-Cyclone	0.4
Landsky	0.3
Ingersoll -Compressor	0.3
PBA	0.2
Other equipment	1.9
Total mechanical supply	**17.2**

14.3.2 TSF & WSF

The TSF and WSF costs have been developed by Golders. The table below details the capital and operating costs associated with the TSF and WSF:

Table 14-5 Total TSF and WSF costs

TSF & WSF Costs	US$ Millions
Initial construction costs - WSF	1.8
Initial construction costs - TSF	1.7
Sustaining capital	1.5
Closure costs	1.2
Total TSF & WSF costs	**6.2**

14.3.3 Project Infrastructure Capital Costs

Project infrastructure costs have been detailed in Table 14-6 below.

Table 14-6 Project infrastructure capital costs

Infrastructure costs	US$ Millions
Bulk power (60 kV power line)	7.9
Roads	1.2
Buildings	1.2
Bulk earthworks	0.8
Overland piping	0.5
Sewage system, run-off and spillage control	0.2
Other infrastructure	0.2
Total infrastructure costs	**12.0**

14.3.4 Mining Capital and Sustaining Capital

Total mine capital and pre-production operating costs are presented below.

Table 14-7 Mine capital cost and sustaining cost summary

Mining capital summary	Capital US$ Millions	Capital US$/Ore t	Sustaining US$ Millions	Sustaining US$/Ore t
Mine capex development	8.9	1.27	62.1	8.86
Mine infrastructure	3.2	0.47	4.1	0.59
Total mine capital	**12.1**	**1.74**	**66.2**	**9.45**

Mining capital costs are based on and includes the following:

- The operating basis is contractor mining
- Fixed infrastructure costs are based on costs supplied by both the contractor and third party suppliers and estimates.
- Site enabling infrastructure, including but not limited to earthworks, site preparation, portal construction, roads, waste dump preparation, communications infrastructure were included in the capital cost estimate both in the contractor’s scope of works and through third parties.
- All underground infrastructure including services supply, dewatering and pumping
- All maintenance costs associated with the maintenance of infrastructure during steady state
- All bulk supply costs associated with mining at steady state
- All costs associated with the owners’ labour team at steady state
- No provision for escalation has been included.

Mine Capital Breakdown

The mine capital and sustaining cost breakdown is presented in

Table 14-8 Mine Capital Cost and Sustaining Costs Breakdown

Mine capital cost breakdown	Capital US$ Millions	Capital US$/Ore t	Sustaining US$ Millions	Sustaining US$/Ore t
Preparatory costs	3.1	0.44	-	-
Lateral development	4.7	0.67	47.5	6.78
Vertical development	0.1	0.02	4.7	0.67
Capital haulage	0.3	0.04	4.6	0.65
Fixed power	0.2	0.02	1.9	0.28
Fixed labour and overheads	0.4	0.06	2.0	0.29
Dayworks provision	0.1	0.02	1.3	0.19
Total mine capex development costs	**8.9**	**1.27**	**62.1**	**8.86**

Mine Infrastructure Breakdown

The mine infrastructure and sustaining capital breakdown is presented Table 14-9 below:

Table 14-9 Mine infrastructure capital and sustaining capital costs

Mine infrastructure breakdown	Capital US$ Millions	Capital US$/Ore t	Sustaining US$ Millions	Sustaining US$/Ore t
Electrical & communications fixed plant	0.7	0.10	1.3	0.19
Ventilation	1.1	0.16	0.1	0.01
Dewatering	0.3	0.04	0.7	0.10
Compressed air	0.2	0.03	-	-
Safety	0.5	0.07	2.0	0.29
Others	0.4	0.06	-	-
Total mine infrastructure	**3.2**	**0.46**	**4.1**	**0.59**

14.3.5 Pre-production Costs

Pre-production costs breakdown is presented in Table 14-10 below:

Table 14-10 Break down of pre-production operating costs

Pre-production operating costs	US$ Millions	US$/Ore t
Pre-production mining costs	1.8	0.25
Pre-production administration and overheads		
Labour	2.4	0.34
General overheads	3.2	0.46
Operating costs	0.9	0.14
Total pre-production operating costs	**8.3**	**1.19**

14.3.6 Closure Costs

Atlas Tin has developed a conceptual mine closure plan based on first principles. The mine closure plan includes the costs of clearing and removing and making safe the following items;

- processing plant
- mine workings and access
- tailings storage facility (TSF)
- water storage facility (WSF)
- waste dumps
- roads and access
- offices and camp
- power lines
- employee redundancies (in accordance with Moroccan labour laws).

A total cost of closure has been estimated at US$3.1 million.

15 OPERATING COST ESTIMATE

As part of the process of developing the Project into a functioning mine, Lycopodium ADP was engaged by Atlas Tin to develop the operating cost estimate (opex) for processing. Entech Mining Consultants was engaged to develop detailed mine physical schedules, which were provided to mining contractors to develop mining opex on a schedule of rates basis.

Certain opex estimates have been sourced from local Moroccan and international suppliers.

These opex estimates were based on 2018 prices.

15.1 Estimate Accuracy

For the scope of work covered and the estimating methodology applied, the opex estimate as presented herein, has an accuracy provision of +/- 15%,

15.1.1 Estimate Base Date

The estimate base date for the definitive feasibility study is 1 April 2019.

15.1.2 Definition of Operating Cost

Operating cost is defined as costs associated with the project that do not form part of the capital cost estimate, pre-production costs and sustaining capital.

Operating cost therefore comprises:

- all mining costs during steady state production and includes:
  - all ore mining costs, excluding underground waste development
  - the cost of stoping, ore drive development and slot raising.
- all processing costs from the start of ore processing
- all maintenance costs associated the with the maintenance of infrastructure and plant split on a ratio between operating and sustaining
- the supply of all chemicals and reagents and power for the processing and site infrastructure
- all bulk supply costs of power, diesel and explosives attributed to ore production and the ore drive development during steady state mining
- all costs associated with the owner's labour team attributed to ore production and the ore drive development during steady state mining during steady state production.

15.1.3 Exchange Rates

The project OPEX estimate is reported in United States Dollars (US$), with all other currencies converted to US$ at the exchange rates in Table 15-1 below.

Table 15-1 Exchange rates

Currency	Rate of exchange to USD
AUD	0.745
ZAR	0.075
EUR	1.157
MAD	0.105

15.1.4 Operating Cost Summary

The target accuracy of the operating cost estimate is -15%/+15%. The average annual operating cost estimate and average LOM unit costs for the project are summarised below in Table 15-2.

Table 15-2 Project summary operating costs

Unit Costs	US$ Million	US$/ tonne Sn Recovered
Sn Recovered		**44,512**
Mining costs	216.6	4,866
Processing costs	109.8	2,466
Administration costs	36.7	824
Concentrate transport and treatment	45.4	1,021
C1 cash costs	**408.5**	**9,176**
Depreciation and amortisation	165.4	3,815
C2 costs	**573.9**	**12,991**
Royalties	25.2	566
Corporate costs	6.1	138
C3 costs	**605.2**	**13,695**
C1 cash costs	**408.5**	**9,176**
Royalties	25.2	566
Sustaining capex	69.2	1,554
Corporate costs	6.1	138
All in sustaining cash costs	**509.0**	**11,435**

15.2 Mining Costs

Mining costs estimated for steady state production are presented in Table 15-3 and include:

- All ore mining costs, excluding underground waste development.
- The cost of stoping, ore drive development and slot raising.
- All bulk supply costs of power, diesel and explosives attributed to ore production and the ore drive development during steady state mining.
- All costs associated with the owner’s labour team attributed to ore production and the ore drive development during steady state mining during steady state production.
- Costs have been developed from contractor unit rates produced from detailed physical schedules, the development of ground support, power and explosives through first principles and process and quotations from 3rd party suppliers for surface haulage and key mine infrastructure.

Table 15-3 Mine operating costs

Mining costs	US$ Millions	US$/Ore t
Lateral development	46.1	6.57
Ore production	56.6	8.07
Haulage	74.5	10.62
Fixed power cost	10.9	1.55
Fixed labour and overhead costs	18.7	2.67
Grade control	5.5	0.78
Others	4.3	0.62
Total operating costs	**216.6**	**30.88**

15.2.1 Key Operating Cost Inputs

Fuel

The cost of fuel has been costed through Afriquia SMDC (Société Marocaine de Distribution de Carburants) an established Moroccan fuel and diesel distributor. Fuel has been costed in the feasibility at 8.78 MAD/ litre (0.92 US$/litre).

On signing a supply agreement Afriquia will provide onsite bunded mobile storage facilities.

Grade Control

Diamond grade control drilling has been included into the mine schedule. Atlas Tin has sourced pricings from Geosond-Maroc, an experienced drilling company in Morocco. Atlas Tin has developed a drilling schedule including rig mobilisation, rig movements and standby costs. This has resulted in an overall cost of grade control diamond drilling of 584 Dh/drill metre (US$ $61.38 /drill metre).

Allowances have been made for power and assaying costs.

Surface Haulage

Surface haulage will be provided by a third-party contractor. During production life of the eastern portal ore and waste will be hauled to the processing crusher pad and the waste dump respectively.

The eastern portal bench remains adjacent to the processing crusher and ore will be direct tipped by the underground mining contractor. Waste from underground will also be able to be stockpiled on the portal bench awaiting rehandling for underground backfill.

The ore sorter will generate crushed rejects material (waste). This material will be direct loaded from an overhead bin into a small truck and relocated to either the waste dump or to the mine portal bench to be for underground use as backfill.

Surface haulage costs (Table 15 9) have been costed where possible based on haulage distances to the ROM pads and waste dumps respectively.

Table 15-4 Surface haulage costs

Destination	US$ / Tonne
Eastern portal to waste dump	0.83
Eastern portal to ore pad	0.62
Central portal to waste dump	0.32
Ores sorter rejects to waste dump / rom pad.	0.30

Explosives

Atlas Tin will provide explosives to the mining contractor. Atlas Tin has sought pricings from EPC-Maroc, one of the world's leaders in industrial explosives and services delivery to the mining industry, to supply and deliver explosives to the project site.

Explosives costs have been costed on a first principles basis. EPC-Maroc will also perform the site compliance and auditing function to ensure compliance to our explosives license.

Ground Support

Ground Support has been developed from quantities developed form geotechnical modelling and design parameters that dove key physical quantities for ground support costing. This included:

- Rock Bolting
- Cable Bolting
- Meshing
- Shotcrete

Power

Grid power costs have been developed from first principles in kWh and applied to an indicative $/kWh unit cost of US$0.09/kWh.

15.3 Processing Operating Costs

Processing operating costs presented herein, has an accuracy provision of -15 to +15 %, representing costs that are omitted or unforeseen.

Costs have been developed from the design flow sheet and associated assumptions interpreted from the project metallurgical test works and associated modelling and experience including project estimates. Costs have been developed from prices sourced from reputable suppliers and manufacturers of chemicals, reagents, infrastructure, mobile plant and equipment.

A summary of processing operating costs is presented below in Table 15-5 as total spend in US$ and US$ per tonne of ore.

Table 15-5 Processing Operating Cost Summary

Processing Operating Cost	US$ Millions	US$/Ore t
Labour	15.2	2.17
Power	31.1	4.44
Ore sorter	0.6	0.09
Regents and consumables	45.2	6.44
Maintenance spares	10.7	1.53
Technical services	5.8	0.83
Import duty	1.0	0.14
Total processing costs	**109.8**	**15.65**

Crushing Plant Loader

A Caterpillar 950 L Wheel loader is included to feed the main crusher and operate the ROM pad located adjacent to the eastern portal. Fuel for the Caterpillar 950L wheel loader has been assumed at 13.2 ltr/hr operating.

Assaying

The processing plant will have an on-stream analysis system to monitor and assist in the control and operations of the plant. An onsite laboratory, 3rd party service provider will be sourced to perform additional site assaying requirements. Atlas management has received a quote from Minerals Processing Solutions, a Moroccan based company which has provided a pricing of 44 Dh/assay (US$4.62) to perform site assay requirements.

A sampling schedule has been developed to determine the number of processing plant assays required for managing the processing plant, concentrate shipments and grade control. Annual cost estimates are provided in Table 15-6 below.

Table 15-6 Assaying costs per annum

Sample Location	US$ Millions pa
Concentrate sampling	0.2
Metallurgical testing	0.4
Environmental analysis	0.1

Concentrate Haulage

Tin concentrate will be trucked to the port in Casablanca. Trucking from Achmmach to Casablanca port, port formalities, customs clearance, and terminal handling fees have been costed on a FOB (free on board) basis. 15 tonnes of concentrate will be shipped in each 20-foot container.

Power

Grid power costs have been developed from first principles in kWh and applied to an indicative $/kWh unit cost of US$0.09/kWh.

15.4 Administration and Overheads

Administration and overhead operating costs are presented in Table 15-7.

All administration and overhead costs are costs expensed during steady state production and includes all costs associated with the owner’s administrative and operational management team.

Table 15-7 Administration and overhead costs

Administration and overhead costs	US$ Millions	US$/Ore t
Labour	12.4	1.77
General overheads	19.4	2.76
Operating costs	3.0	0.43
Power	1.9	0.27
Total administration and overhead costs	**36.7**	**5.23**

16 FINANCIAL ANALYSIS AND EVALUATION

16.1 Introduction

The Achmmach Tin Project is jointly owned by Kasbah Resources Limited (75%), Toyota Tsusho Corporation (20%) and Nittetsu Mining Corporation (5%). All the numbers presented in this financial evaluation section of the report are based upon 100% ownership.

Furthermore, the project is evaluated on a 100% equity basis only and excludes any financial leveraging effects (i.e. ungeared), as well as any interest expense items that could impact taxable income and/or provide interest deduction tax shields.

The analysis is based on a processing throughput of 750 ktpa using the Ore Reserves defined by this feasibility study. The Ore Reserves Estimate is based on a 0.55% Sn cut-off grade.

The objective of the economic analysis is to demonstrate the economic viability of the Project, provide support for project financing activities, and enable the shareholders to reach formal decisions to proceed with the detail design and construction phases of the project.

16.2 Evaluation Methodology and Analysis

16.2.1 Evaluation Methodology

The financial evaluation has been performed using real cash flows. No escalation of cash inflows or outflows has been applied in determining the project NPV and IRR.

The evaluation is based on after-tax unleveraged, real internal rate of return (IRR) using monthly cashflows using mid period convention.

Exchange rates used in the project cost analysis and evaluation are detailed below in Table 16-1 Project exchange rates. These exchange rates have been determined based on spot prices dated 2 July 2018.

Table 16-1 Project exchange rates

Currency	Rate of exchange to USD
AUD	0.745
ZAR	0.075
EUR	1.157
MAD	0.105

16.2.2 Base Date

The valuation date for the financial model is 1 April 2019, reflecting when the project will be funded for development.

16.2.3 Revenue

On-mine revenue is derived from the sale of tin concentrates into the international market place. Revenues are calculated by the tin content of the concentrate less unit deductions, smelting and refining and impurity penalty charges. No marketing fees have been reflected in the model as Atlas

Tin expects to enter into long term offtake agreements with one or more traders/smelters with FOB shipping terms, where ownership and control ceases once loaded at the port of Casablanca.

16.2.4 Tin Price Forecast

For purposes of financial modelling, a price of US$21,000/t (real 2018 $) has been used. This price reflects current spot price and has been determined based on the 105-month historical average plus 15-month LME futures contract.

Further analysis on the tin price has been included in Section 13. Sensitivity of tin price to project economics have been analysed below.

16.2.5 Marketing and Treatment Charges

Through a market soft-sounding, Atlas Tin gained indicative non-binding offtake terms for its tin concentrate from a number of traders and smelters to established indicative net smelter returns (NSR) and average metal pay ability primarily for financial model purposes. Concentrate and treatment charges have been applied which include penalties and unit deductions for impurities.

Table 16-2 Concentrate specification

Element/Compound	Specification
Sn	60.00%
Fe	4.01%
Mn	0.02%
WO3	0.05%
Pb	0.03%
Zn	0.02%
Ni	0.00%
Co	0.00%
Ag	1.92 ppm
Cu	0.00%
As	0.08%
Bi	0.00%
Sb	0.04%
S	1.73%
ThO2+U3O8	0.00%
F	0.09%

16.2.6 Production and Revenue Summary

Construction of the project is scheduled to take 18 months, before first production in the second half of 2020. A six-month production ramp-up has been scheduled for commissioning of the plant before achieving full design capacity.

16.3 Revenue Inputs and Assumptions

Revenues are calculated in United States dollars based on a metal price of US$21,000/t (real 2018$). Penalties and unit deduction charges are deducted in calculating the on-mine revenue.

The financial model assumes that 90% of the revenues are received in the month following production, with the balance received 3-months later.

16.4 Project Capital Costs

The project execution capital costs summarised in Table 16-3 have been estimated as US$96.4 million (real 2018 $).

Table 16-3 Project capital costs

Capital cost	US$ Millions
Mining development	12.1
TSF and WSF	3.5
Process plant	44.5
Infrastructure	12.0
Engineering Procurement and Construction (EPC)	7.2
Construction indirect costs	5.4
Sub Total – Project Construction Capital	**84.7**
First fills and spares	1.2
Contingency	10.5
Total Project Capital Costs	**96.4**

16.5 Sustaining and Mine Closure Capital

Sustaining (or replacement) capital has been estimated into the operating costs for the mining and processing operations.

Sustaining capital for the site infrastructure has been estimated at US$69.2 million over the life of mine. The mine closure cost has been estimated to be US$3.1 million. Salvage values from dismantling and sale of the process plant has been estimated at US$2.7 million.

16.6 Operating Costs

The total operating costs are summarised in Table 16-4 below.

Table 16-4 Project operating costs

Unit Costs	US$ Million	US$/ tonne Sn
Mining costs	216.6	4,866
Processing costs	109.8	2,466
Administration costs	36.7	824
Concentrate transport and treatment	45.4	1,021
C1 cash costs	**408.5**	**9,176**
Depreciation and amortisation	165.4	3,815
C2 costs	**573.9**	**12,991**
Royalties	25.2	566
Corporate costs	6.1	138
C3 costs	**605.2**	**13,695**
Sustaining capital	69.2	1,554
All in sustaining cash costs (AISC)	**509.0**	**11,435**

C1 cash costs consist of mining, processing, administration and concentrate transport & shipment.
AISC consist of C1 cash costs, royalties, corporate overheads and sustaining capital

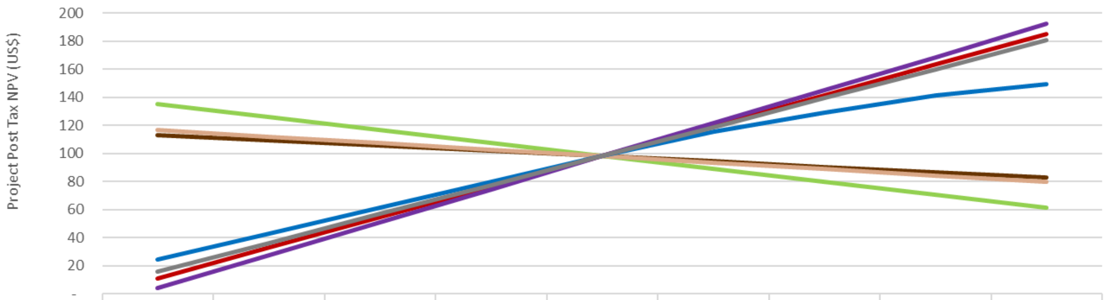
Table 16-5 Financial summary

Parameters	
Sn Price	US$21,000/t
Discount rate	8%
NPV, post tax[1]	$98.1M
IRR, post tax[1]	23%
Capital costs	$96.4M
C1 cash costs	**$9,176/t**
C3 costs	**$13,695/t**
Average AISC	**$11,435/t**
Operating cash flow	$403M
Free cash flow	$267M
Turnover	$815M
EBITDA	$444M

[1] Project NPV and IRR based on a post-tax discount rate of 8% and Moroccan Corporate Income Tax of 17.5%

16.7 Sensitivity

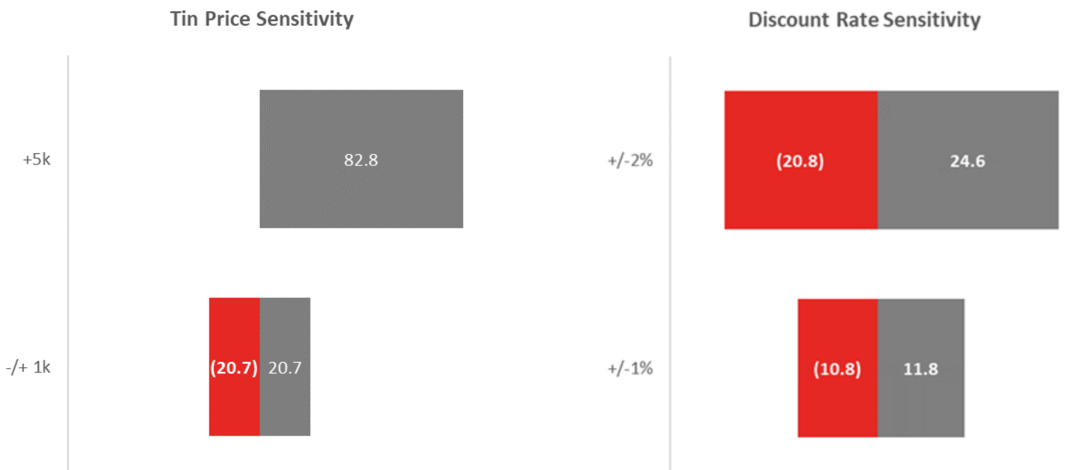
The Project economics are most sensitive to changes in the tin price, grade and recovery. The sensitivity of the NPV to changes in major cost and revenue drivers is shown in Figure 16-1 below.



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Sn Price	11	33	55	76	98	120	142	163	185
Capital Costs	113	109	106	102	98	94	91	87	83
Operating Costs	135	126	117	107	98	89	80	70	61
Exchange Rate	117	112	107	103	98	93	89	84	80
Throughput	24	43	61	80	98	115	129	141	150
Grade	4	27	51	75	98	122	145	169	192
Recovery	15	36	57	77	98	119	139	160	181

Figure 16-1 Project NPV sensitivity

Detailed NPV and IRR sensitivity analysis was performed on the major cost and revenue drivers and the NPV analysis is presented for the individual factors in Figure 16-2.



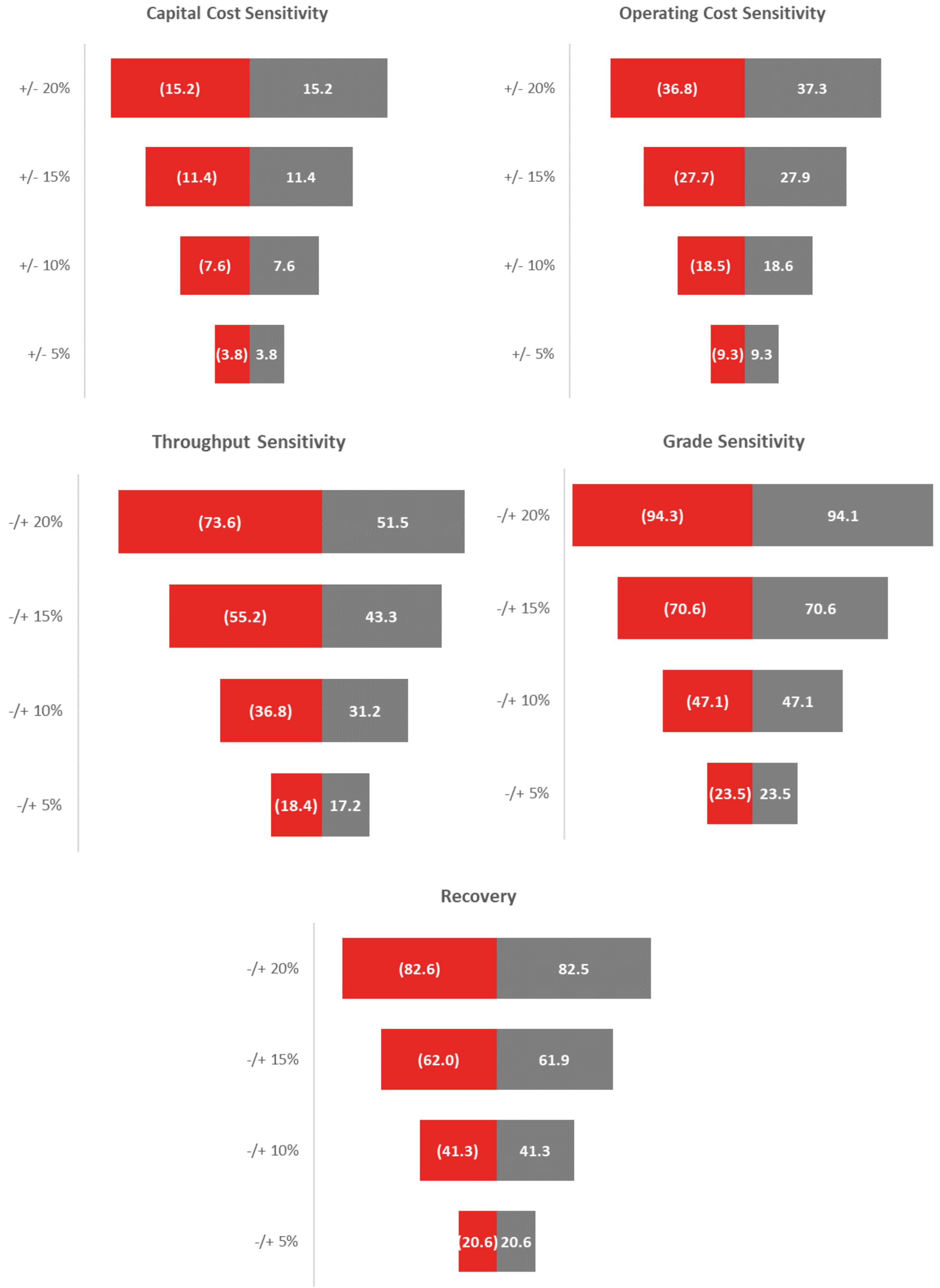


Figure 16-2 Detailed sensitivity analysis

16.8 Mine Life

The operating mine life of the Achmmach Tin Project, based on the assumptions in the Achmmach Definitive Feasibility Study Report, is 10 years. This includes the initial production ramp-up but excludes project construction, pre-production and mine closure activities.

Extending the life of the mine would rely on the conversion of measured and indicated resources to reserves and the discovery of additional resources within the current mining permit area. The existing ore bodies are open along strike and at depth providing excellent exploration potential.

17 PROJECT EXECUTION

17.1 Introduction

This section describes the project execution strategy for the project development from commencement to handover of the operating facility to the operations team. The costs associated with this strategy are included in the capital estimate in Section 14.

The project execution plan (PEP) has been developed by Lycopodium ADP in consultation with the Atlas Tin. The PEP details what the project will deliver and how it will be delivered to handover of the operating facility to the Atlas Tin operations team.

The PEP addresses the following:

- project scope
- contracting strategy
- resourcing and organisation
- engineering and detailed design
- procurement and logistics
- construction and construction management
- mine development
- project controls
- project schedule
- health, safety, environment and community
- commissioning, handover and operational readiness.

This section is a summary of the key aspects of the PEP which is based on an EPC approach. The detailed PEP will be created during the next phase of the project.

17.2 Development Methodology

The objective of the project execution strategy is to provide Atlas Tin with a technically successful mining and processing operation that is brought into operation as soon as possible on a cost-effective basis.

Technical success will result in a processing plant that is able to ramp up to full production capacity in as short a time as possible. This will be achieved through a detailed understanding of the project flowsheet and proper selection of equipment to achieve the process requirements. The adherence to the project schedule will be achieved through sound planning, particularly in relation to the critical path activities. An EPC Contractor will be engaged to design and construct the plant and associated infrastructure on a turnkey basis. This will provide contractual obligations relating to technical success, time and cost.

Atlas Tin will establish a project management team (owner's team) to manage the total project scope which can be divided into three major areas:

- design, construction and commissioning of the processing plant and associated infrastructure
- development of the underground mine and its associated infrastructure
- development of the TSF.

17.2.1 Project Scope

The scope for the development of the project is separated into the following major areas:

- design and construction of the processing plant and related infrastructure
- design and construction of the water storage facility and tailing storage facility
- offsite infrastructure, such as access road repairs, ONEE power line, etc. to be performed by selected contractors
- the development of the underground mine and associated infrastructure
- establishment of an operations team, to run and maintain the various aspects of the operations as they are implemented.

All these activities are continuations of work performed to date by Atlas Tin and its consultants during the course of the feasibility study. Each of these activities is briefly described below.

Design and Construction of Plant and infrastructure

It is planned that an EPC Contractor will be appointed to perform the design and construction of the processing plant and associated infrastructure. The capital cost estimate prepared for the processing plant makes suitable allowance for EPC fees and contingencies.

The execution strategy for the processing plant and infrastructure is discussed below in Section 7.

Design and Construction of Water Storage Facility and Tailings Storage Facility

Golder (engineering consultants) has provided designs to feasibility study standard for the water storage facility (WSF) and tailings storage facility (TSF). The next stage will see final design of the WSF completed during the financing period, such that the construction of the WSF can commence as soon as project funding is available. This is to allow maximum harvesting of water prior to plant start-up.

Allowance has been made for Golder to perform the design work, as well as to provide construction supervision to confirm construction technical compliance. The construction works will be performed by an earthworks contractor with the design consultant supervising to ensure technical compliance.

The design and construction of the TSF will occur during the project execution phase, post funding.

Offsite infrastructure and other items

There are various items of offsite infrastructure, such as access road repairs and powerlines that will be let to various local contractors to provide design/construction/installation contracts.

House /property purchases; as detailed in Section 9, Atlas Tin is obliged to purchase a number of properties prior to the construction and/or operation of the plant.

Development of the underground mine and associated infrastructure

A mine development and production contract will be awarded to a suitably qualified contractor to develop the underground mine. The final mine layout has been established as part of this feasibility study. The portal design will be completed by a mining consultant (Entech) during the first three months of the project execution phase.

Further details of the mine development are described in Section 5 of this study report.

Atlas Tin will manage the prequalification and tendering of the mine services contract with the assistance of a specialist mining consultant.

The management of the mining contractor activities will be handed over to the mining manager. Costs have been allowed for Atlas Tin's mining consultants to assist in knowledge transfer to the mine manager (ie. explanation of mine plans and schedules). Costs have been allowed in the capital and operating costs for the mining contractor's full scope including the construction of haul roads, plant waste for mine backfill, and the ongoing construction of the TSF embankment.

Establishment of Operations Team

Atlas Tin will establish a fully resourced operations team on a progressive basis to take ownership of and manage the various project elements as they are developed. This is discussed further in Section 8.

17.2.2 Atlas Tin Project Management Team Structure.

To manage the development of the project Atlas Tin will utilise a mix of project specific personnel and consultants, together with operations team members to deliver the project elements as outlined in the organisational structures in Section 8.

Management of project elements

- An EPC contractor will be engaged for the design and construction of the processing plant and related infrastructure. This work will be managed by the project manager
- A specialist consultant will be engaged to design and supervise the construction of the WSF and TSF. An earthworks contractor will be engaged to build the earthen structures. This work will be managed by the project manager. The latter stages of the TSF construction will be performed during the life of the mine and will be managed by the mining department.
- Offsite infrastructure, such as access road repairs and the power line will be performed by selected contractors. This work will be managed by the project manager.
- All work related to the design and construction of the mine will be managed by the general manager, and then delegated to the mine manager once recruited. The project manager will assist as required for any surface infrastructure to be provided by the EPC contractor.
- The establishment of the operations team will be the main task of the general manager.

Owner activities will be mostly managed from Perth, until the majority of activities are underway at site, whereupon the team will relocate to site/Meknes.

17.2.3 EPC Contractor Scope Activities

The processing plant and required infrastructure are described in Section 6 of this study.

The processing facilities are:

- technically complex
- multi-discipline
- design intensive and have a detailed interlinked schedule.

As such, implementation of the project contributes significantly to the technical risk and by nature of the complex schedule interconnections, a large proportion of the project's schedule risk.

The infrastructure portion of the project consists of several essentially stand-alone tasks which are:

- technically less complex than the processing plant
- generally single discipline (e.g. earthworks or buildings or electrical reticulation)
- have independent and relatively straight forward schedules
- suitable for vertical construction or design and construction contracting strategies.

17.3 Project Execution Scope

The project scope outlines the complete list of deliverables that comprise the project. The scope of the project includes the mine development, process design, engineering, detailed design of the facilities, all construction management, commissioning and operational readiness to enable mine and plant production to be achieved.

The project is of such a size to enable it to be divided into discrete and specific areas of responsibility and assigned to selected managers. The responsible managers will work collaboratively to ensure successful delivery of the project.

17.3.1 Owners Scope of Work

The activities to be directly managed by Atlas Tin are:

- the management of the plant and infrastructure EPC contractor
- the development of the underground mine and associated haul road
- the design and construction of the tailings management facility (TSF)
- the design and construction of the water storage facility (WSF)
- overhead 60 kV powerline
- environmental permit approvals
- environmental baseline monitoring
- property purchases
- operations staff recruitment
- access road upgrade
- other owner activities.

These activities are continuations of work performed to date by consultants and Atlas Tin during the feasibility study.

17.3.2 EPC Contractor's Scope of Work

Broadly, the following major areas are considered to be part of project execution scope for the EPC contractor:

- site project management
- bulk earth works and in plant roads
- processing plant
- process plant buildings
- outdoor switchyard (connecting to incoming 60 kV power line supplied by vendor)
- mine services area
- temporary construction village (and associated services) which ramps up to accommodate 72 people at peak (Atlas Tin and EPC contractor teams) which will convert to a permanent operational village for 51 people
- communications for camp and process plant site

- stores
- HV and LV workshops.

17.3.3 Mine Development

The study mine design and schedule will for the basis of the scope and specification for the contractor. The process for tendering is as follows:

- pre-qualification of suitable contractors
- tendering, and award of a mining services contract
- mobilisation of the selected contractor at which point site facilities and workshops will be established by mining contractor and the EPC contractor.

The prequalification and tendering of the mine services contract will be managed by Atlas Tin with the assistance of a specialist mining consultant.

The management of the mining contractor will be handed over to the mining manager, following their recruitment.

17.3.4 Tailings Management Facility and Water Storage Facility

Golder has provided designs to a feasibility study standard for the TSF and WSF. The next stage will see the final design and tender documents completed. An allowance has been made for suitably qualified consultants to perform this work which is scheduled to be performed early in the execution phase to allow the design to be fully completed prior to work on site. The consultants scope will include construction supervision and the establishment of a soils testing laboratory for QA/QC.

17.3.5 Property Purchases and Resettlement

As detailed in Section 10, Atlas Tin is obliged to purchase a number of properties prior to the construction and/or operation of the plant. Estimated costs for the purchase of the properties have been included in the cost estimation.

17.3.6 Operations Staff Recruitment

Atlas Tin plans a progressive ramp-up for recruitment of operational personnel. The salaries of recruited personnel prior to commissioning, together with the recruitment costs of the operations team have been estimated and allowed in the owner’s costs estimate. The recruitment process will be managed by the general manager.

17.4 Contracting Plan

17.4.1 Major Contracts

The following major contracts will be managed by the owner’s team:

- design and construction of the plant and infrastructure (EPC Contract-further described Section 17.6)
- mining services (mining contract).
- design and construction supervision of the TSF and WSF
- permitting, design and construction of the powerline
- provision of laboratory services and equipment for the mine and processing plant operation.

Suitably experienced legal practitioners will be engaged to draft relevant terms and conditions.

17.5 Minor Contracts

The owner's team will manage the following service contracts:

- metallurgical consulting
- engineering consulting
- environmental monitoring
- hydrogeology consulting
- mining engineering consulting

Relevant terms and conditions will be determined by suitably experienced legal practitioners.

17.6 EPC Contract

The EPC contractor will provide the following services associated with the development of the Achmmach project:

- finalise geotechnical engineering for the plant site, site and plant access roads
- design and construction of plant power distribution system including coordination with site power supply authority
- design and installation of project buildings
- surveying services during construction
- process engineering
- design engineering and drafting of earthworks, civil works, structural steel, mechanical and electrical installations
- project management services including cost control, scheduling, reporting and claims processing
- procurement of materials, equipment and fabricated items including tendering, purchasing, expediting and contract preparation and administration
- logistics coordination
- construction management including site management, and control and inspection of all construction activities
- commissioning including pre-commissioning and testing, dry commissioning, wet commissioning and operational assistance until handover
- regulatory compliance monitoring and reporting of project activities
- overall site management until completion of commissioning.

These specialist services will be provided by the EPC contractor's staff or will be provided to the engineer on subcontract arrangement. The EPC contractor will execute most of the engineering, design and procurement from its head office and will maintain direct construction management from the site office at the plant site. The execution strategy recognises the requirement for sign-off of various designs by authorised personnel to meet statutory requirements. This will be achieved by engaging suitable individuals through Moroccan consultants to input to, review, and sign-off. Key members of the Atlas Tin project team will be regularly resident in the engineer's office during detail design to review and approve engineering design and equipment and materials selection. These personnel will then relocate to site during construction.

Atlas Tin will retain responsibility for government liaison and permitting, all operating and related activities, community relations, asset insurances, land purchases and legal services.

17.6.1 Project Duration

The project schedule for the project is appended to this section of the report (Appendix 17A). The schedule indicates the following timing for key activities associated with the development of the project.

Table 17-1 Key project milestones

Milestone	Date
Project commencement & site construction	30 April 2019
Administration start date	31 October 2019
UG site mining contractor – site establishment	29 February 2020
Boxcut establishment	31 March 2020
Grid power energised	31 May 2020
UG mining commences	30 June 2020
Completed plant & surface facilities	30 November 2020
Processing commences	30 November 2020
Processing end date	31 December 2030

Atlas Tin will continue to review and optimise the project timing and schedule. Management is continuing to explore ways to reduce the project construction timing.

17.7 Engineering Design

Detailed engineering design of the processing infrastructure would commence immediately after the award of the contract for the design and construction of the plant.

17.8 Equipment Procurement

There are several items of long lead equipment associated with the processing facilities. Some long lead equipment packages include:

- crushers
- ore sorter
- HPGR
- ball mill
- motor control centres and switchrooms
- transformers
- concentrate filters.

The schedule provides for 50 weeks for the procurement of major equipment. This activity is indicated commencing eight weeks after the commencement of design activities to enable the confirmation of equipment requirements prior to ordering.

17.9 Bulk Materials Procurement

Bulk materials procurement will include:

- concrete reinforcing steel
- tank and bin platework
- fabricated and surface treated structural steel
- floor mesh and hand railing
- pipes, valves, pipe fittings and supports systems
- electrical cable, cable tray, lights and miscellaneous equipment.

These items will be progressively specified, purchased, expedited and transported to site by the project team as required.

17.10 Site Works Contracts

The EPC contractor would qualify suitable Moroccan or European based construction organisations to perform appropriate site works. Companies contacted during the detailed feasibility study will be used to assist in this process. The site works would be divided mainly into horizontal discipline packages to make the best possible use of individual subcontractor capability, capacity, organisation skills, expertise, and personnel resources.

Typical subcontractor packages to be adopted will be:

- earthworks, water dams and road works
- concrete civil works
- structural installations (including rigging)
- plant buildings installation
- general mechanical installations
- specialised mill installation
- process plant piping installation
- site HV powerline installations
- process plant electrical and instrumentation installation
- water and tailings lines installations including water pumps
- construction equipment hire
- temporary construction facilities hire.

The general scope for fabrication would include elements to supply, fabricate, surface treat, pack and prepare for transport, catalogue and load onto transport. The EPC contractor would specify the required surface treatment and approve the application of the paint or lining system. The selection criteria for the fabrication subcontractors would include a combination of reputation, quality, safety, location, workshop capacity, commitment and performance to schedule and price.

17.11 Health and Safety

17.11.1 Health and Safety Principles

All aspects of the project will be required to comply with relevant local legislation and current international industrial practice. The project's principal health and safety objective is to develop a culture and install processes to ensure the safety and health of all employees, contractors and stakeholders. In striving for this objective, the following principles will apply:

- No business objective will take priority over health and safety.
- All contractors shall demonstrate commitment to high health and safety standards to be eligible to work on the site.
- At all times, the safety of all personnel is to be ensured.
- The project will target zero harm.
- All incidents and injuries shall be reported, investigated, and actions taken to prevent reoccurrence.
- Accountability for providing a safe work environment rests with every employee.
- All individuals have the responsibility and accountability to identify and eliminate or manage hazards / risks associated with their workplace.
- All employees (including those of contractors) are to be trained and equipped to have the skills and facilities to enable an injury free workplace.
- All employees (including those of contractors) are to undergo project specific inductions as appropriate to their site work requirements.

Each individual participating in the project will be expected to actively contribute to the promotion of constructive health and safety practices and in the implementation of health and safety management plans.

17.11.2 Health and Safety Management Plan

A project specific health and safety management plan will be prepared. The management plan will be issued to all contractors tendering for site work as part of the tendering process. Each contractor will be required to demonstrate a satisfactory prior commitment to health and safety and present their own site specific plan for their proposed involvement in the project which as a minimum must comply with the owner’s health and safety plan and associated requirements.

17.12 Project Controls

Effective project controls are critical to the successful ongoing management and ultimately the successful completion of a project. The controls provide relevant and consistent budget, costs and schedule reporting to the project team. This provides the tools to efficiently manage the project at the level of detail necessary to meet project objectives.

The project controls team shall be responsible for the following:

- project budget management
- cost tracking and reporting
- cash flow forecasts and payment schedule
- reporting and forecasting
- commitment approval process
- risk register
- schedule tracking and reporting
- contract management and administrative tasks on site.

The project controls plan (PCP) addresses cost control, planning, progress measurement, project reporting, asset capitalisation and close-out. The scope of the PCP is to provide a framework of the work processes, work flows and information relating to the standard project controls and accounting interface tools, systems and procedures.

17.12.1 Cost Control

The Work Breakdown Structure (WBS) defines the project in terms of activity level / areas that can be clearly defined, managed and controlled. The WBS will encapsulate the total project scope and defines how the project will be subdivided into smaller, manageable portions for cost control, estimating, budgeting and scheduling purposes.

The capital cost estimate developed during the study will be used as the control budget for the project. Costs will be measured and reported by activity in accordance with the WBS. This will serve to keep the owner informed on a timely basis about risk associated with cost and time. Monthly cost reports will be prepared to show original budget, approved changes, revised budget and current forecast costs. Committed and incurred costs and paid expenditures will also be included in the report.

17.12.2 Planning and Scheduling

Key challenges that will need to be actively managed from a scheduling perspective include:

- being able to drive the project using the schedule as opposed to just tracking progress
- ensuring timely submission by all contractors to meet owner's team reporting requirements
- capturing of minor contract works and self-managed minor works on site.

A full time scheduler is required to commence concurrent with design activities. The principal function of the scheduler is to provide the appropriate information to enable the EPC contractor's and owner's team personnel the means to drive the project, report on progress and more specifically manage the project critical path.

The scheduler will capture progress on a weekly / monthly basis and submit analysis on the overall project critical path, key issues and possible improvements / opportunities.

17.12.3 Project Approvals

A detailed approvals matrix will be developed for the project. The approvals matrix will address:

- expenditure approval levels
- correspondence and issuing of notices
- scope changes
- approvals
- hiring and retrenchments.

17.12.4 Change Management

The purpose of a change control system is to manage and track any changes and the associated costs outside of or different to, the agreed project scope / requirements and may impact any deliverable, system, or project plan. It is also the mechanism to use to track changes that are within the Scope, i.e. system changes to forms or reports.

The change control system is designed to allow the owners team the flexibility to incorporate any agreed changes to owner's scope and the EPC Contractor's scope while retaining control over costs and time frames. Change control will become effective when the baseline for the project is established. The aim is to baseline all budgets and schedules once a formal project commencement date is known.

All changes that have the potential to impact the project end-date will be reviewed against other project priorities / deliverables to identify potential trade-offs. The objective of this review is to protect the project end-date and budget.

17.12.5 Communications

Effective communication and decision making will be a pivotal component of project success. Based on the project structure, overall project external reporting and communication responsibility will rest with the owner's team. A detailed communication plan / matrix will be developed prior to project commencement.

17.12.6 Document Management

The project controls team will be responsible for the receipt, storage, tracking and issuing of all documentation being issued to or by the project. On completion of the EPC contract all documentation including vendor documentation (hard and soft copy) will be issued by the EPC Contractor to the owner's team.

17.12.7 Quality Control - QA/QC

Project construction QA/QC will reflect the execution strategy and scope divisions that have been adopted for the project:

- For the process plant modules, the EPC Contractor in conjunction with the owner's team and primary vendor(s) will develop and implement a suitable QA/QC program broadly compliant to AS/NZ ISO 9001/2008 quality management systems.
- For the site activities, most owner's scope QA/QC will be largely limited to on the job inspections and a much greater emphasis on punch-listing with a fit for purpose approach to be adopted by the owner's team to ensure conformance with design intent. The exceptions to this will be:
    - Tailings and return water facility construction - A third party specialist engineer including a dedicated laboratory will be on site to ensure fully documented construction / design compliance of the tailings storage facility owner's team.
    - Camp electrical installation and testing - All camp electrical works will be completed, tested and documented with regard to QA/QC in line with applicable standards.

17.13 Project Commissioning and Closeout

17.13.1 Strategy

A project specific commissioning plan will be compiled. The plan will address the transition from a construction site to a complete and operational mine that meets design objectives. The process of confirming the objectives are met will be captured by a documented commissioning procedure with appropriate closeout documentation inclusive of a final report.

17.13.2 Commissioning Philosophy Definitions and Handover

Clear hand over points for all parts of the project will be determined at an early point in the project build cycle. A clear philosophy will be adopted that ensures inclusion of as many new and future operations staff in commissioning activities as practical to ensure information transfer and to limit commissioning costs.

17.13.3 Handover Point Philosophy

The following nominal commissioning definitions will be used to properly define handover requirements:

- C1 - Electrically complete with respect all pre-energisation tests and energised / run.
- C2 - Run with tested with water.
- C3 - Tested with water and feedstock not at design rates.
- C4 - Tested with water and feedstock metallurgical design achieved.

The project commissioning will be performed by the vendors, EPC contractor and owner's operations staff. Handover documentation will include:

- commissioning documentation plus manuals
- care and custody sheets for used equipment that didn't require formal commissioning
- up to date electronic copies of all civil, structural, mechanical, electrical and piping drawings
- up to date electronic copies of all vendor drawings
- as built drawings and engineering deliverables as agreed.

A formal closeout report will be compiled at project completion which will provide an overview of the all aspects of the engineering, procurement and construction management of the project.

17.14 Operational Readiness

17.14.1 Aim

Operational Readiness is a key building block of the project and is especially important given that the primary drivers of the project team are:

- To safely deliver a suitable project via a design, construction and commissioning team on budget and on schedule.
- To develop and implement operational readiness plan (ORP) such that when the project is built a team is in place with the necessary spares and stems to operate.
- To develop and maintain a social license to operate the project.

The aim of operational readiness is therefore a smooth transition from project to commission to ramp up and first tin production in the first instance but sustainable production thereafter.

17.14.2 Operational Readiness Scope

All activities associated with operational readiness are in the owner's team budget and scope. Specialist contractors may be engaged directly to assist in this process and a level of cooperation will be built into the EPC contract, to enable interfacing specifically with respect to equipment spares supply and first fill requirements. Operational readiness is broadly anticipated to include:

- recruitment of operations team
- messing and accommodation of operations team
- all operational permits, processes and procedures
- all operational service and supply contracts
- all mining activities and associated contracts
- all site security arrangements and staffing
- all community and social
- all regulatory and reporting.
- operational software and licensing.
- insurances and sales contracts.
- site maintenance strategy.
- process plant lube schedules.
- site wide work orders and preventative maintenance procedures.
- site wide bills of materials.
- site wide maintenance systems upload and implementation.

18 WORK PLAN

18.1 Introduction

Following the satisfactory completion of the DFS in July 2018 and confirming that there is a viable operation to be developed, Atlas Tin has outlined a 9 months period to raise the requisite debt and equity financing, to enable project execution to commence as detailed in Section 19 of the 2018 DFS, Project Execution and Engineering Development.

During this nine months and in parallel with the financing activities, Kasbah has defined a works plan to progress key activities and maintain project momentum, which addresses the following areas

- critical infrastructure
- key operational management plans
- permitting & approvals
- resettlement
- recruitment
- test works
- procurement supply & logistics
- exploration

18.2 Critical Infrastructure

18.2.1 Power Line

The Achmmach project requires the construction of a 60 kV line from the Toulal station, 44 km to the north. This will require formal agreement with ONEE (The Office of National Power and Potable Water). The Achmmach camp site is currently serviced by an 11 kV line that also services the neighbouring mine. This 11 kV line will provide power for construction. The construction of the main 60 kV high voltage line to Achmmach is the longest lead time item in the project construction plan with an expected construction period of 12 months.

The ability to have the project power supply in place prior to commencement of the underground development will remove the need for site-based diesel generated power, further supporting project commissioning and ramp up.

18.2.2 Water Storage Facility (WSF) & Tailings Storage Facility (TSF)

Kasbah engaged Golders to undertake the revised design and capital construction costs of the water storage facility (WSF) and the tailings storage facility (TSF) for the feasibility study. The detailed design will commence during the financing period. This work will be completed by a suitably qualified consultant and will include the detailed drawings, schedule of quantities, tender specifications in preparation for tendering and operating manuals.

18.3 Key Operational Management Plans

The Achmmach ESIA outlines several key management plans that are required prior to commencing mining operations. These are seen as critical to the commencement of construction activities and will underpin the culture and standards of the operation through construction and into operations.

The key management plans that are proposed to be undertaken in the funding periods are:

- environmental safety & health management plan
- community engagement management plan
- grievance management plan
- traffic management plan.

It is proposed that the Moroccan environmental consultant that developed the ESIA will be engaged to support the development of key elements of these management plans.

18.4 Permitting & Approval

Atlas tin requires permits and approvals to support the development of the Achmmach project. These are:

- approval to construct
- water storage dams
- explosives.

During the nine-month financing phase Atlas Tin will progress to obtain all required permits and approvals necessary for project construction and commissioning to commence.

18.5 Resettlement

Once the decision to progress has been made by the board project representatives will commence formal discussions through the Governor's office and the two local commune presidents regarding the resettlement. This will allow the development of the Resettlement action plan to begin.

18.6 Recruitment

Through the financing phase of the project Kasbah will begin the process of identifying key management personnel and continue to develop and build the understanding and capacity within the local and wider regional population in Morocco. This would include meeting with local recruitment services and advertising in newspapers seeking expressions of interest. This will build up an understanding of the local workforce.

18.7 Test Works

During the 2018 test works program a number of additional process improvement opportunities were identified. These additional opportunities will continue to be explored to confirm their suitability for implementation and inclusion through the final engineering and design phase. In addition to the metallurgical and processing design improvements, geotechnical trenching and mapping of the revised mill site location is required to finalise site earthworks requirements and costs.

These works will be performed and finalised during the financing period to support the ability to rapidly transition to the final engineering and design phase.

18.8 Procurement Supply and Logistics

18.8.1 Pre Tendering

During the financing period management will continue to identify service providers for the key Achmmach contracts: -

- EPC contract
- Underground mining services

Management intends to identify screen and short list key these providers

18.8.2 Tender documentation

Potential EPC and Mining Contractors will be identified. Draft contracts will be developed with assistance of Kasbah Legal advisors. Contracts will be tendered and adjudicated ready for award subject to Kasbah board approval, and stakeholder (funders) agreement.

18.8.3 Supply & Logistics

Atlas Tin continues to explore and develop an understanding of the supply, logistics and support industries available within Morocco. The ability to access the local Moroccan supply base is seen as key to the success of the project. Atlas will continue to develop this data base in Morocco to ensure the process of identifying and tendering works is efficient and that the company has a competitive local supply base.

With the 2018 DFS finalised Atlas Tin management can seek to further explore pricing and supply base capabilities within Morocco specific to Achmmach as identified during the 2018 DFS. This includes, infrastructure, chemicals reagents and support services.

19 Appendices

- Appendix 3A: Quantitative Group Consulting – Achmmach Mineral Resource Estimate
- Appendix 4A: Mining One Consultants – Geotechnical Evaluation for Achmmach Project
- Appendix 5A: Entech Mining Consultants – Design and Scheduling
- Appendix 6A: Tony Parry & Associates - Ore Sorting Testwork
- Appendix 6B: Lycopodium ADP - Design Criteria
- Appendix 6C: Mike Gunn - Mass Balance
- Appendix 6D: Lycopodium ADP - Equipment List
- Appendix 6E: Lycopodium ADP - Process Flow Diagrams
- Appendix 6F: Lycopodium ADP - Plant Layout
- Appendix 7A: Golders UK – Technical Report
- Appendix 7B: Golders UK – Geochemical Assessment
- Appendix 9A: Artelia - Environmental and Social Management and Monitoring Plan
- Appendix 9B: Artelia - Environmental and Social Impact Assessment
- Appendix 12A: Joint Venture Agreement
- Appendix 17A: Project Schedule