MODELLING AND EVALUATION OF THE TIN RESOURCE ON TWO FARMS ON THE POTGIETERSRUS LIMB OF THE BUSHVELD COMPLEX

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A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements of the degree of Master of Science in Engineering

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DECLARATION

I declare that this research report is my own, unaided work. Where use has been made of the work of others, it is fully acknowledged in the text. It is being submitted for the Degree of Master of Science in Mining Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before in any form for any degree or examination in any other University.

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Arnold Muindisi

This______________Day of_______________2012

ABSTRACT

The properties of Roodepoort 222KR and Groenfontein 227KR are located within part of the Zaaiplaats tin mining district about 30km north-west of Mokopane (Potgietersrus). Tin mineralisation occurs within the upper part of a sheet of Bushveld granite which overlies the mafic rocks of the Potgietersrus Limb of the Bushveld Complex. Tin has been commercially mined in this area since 1907 and has been recovered by Zaaiplaats Tin Mining Company from Zaaiplaats 223KR, Roodepoort 222KR and Groenfontein 227KR, the latter two having been worked under tribute from Transvaal Consolidated Lands mining company (TCL). In 1978, a major drilling program was carried out with the aim of investigating the economic viability of the remaining resource on Roodepoort and Groenfontein. The programme background, implementation and results were assessed and reported on by I.M. Clementson in February 1979.

The programme was divided into three drilling and sampling phases. The outcome of the first two phases revealed that mineralisation in the upper fine grained Lease Granite is largely restricted to the surface outcrop area of an exposed elongate dome-like structure and only extends for a limited distance down dip. It was observed that the overlying pegmatite zone and pipe ore bodies contain good but isolated and unpredictable tin grades. Twenty boreholes, RDP077 to RDP097, were collared in visibly mineralised Lease Granite within a surface geochemical anomaly and gave encouraging results. This prompted Phase 3 which involved percussion drilling on a 30m by 30m grid pattern over the major portion of the surface geochemical anomaly. Boreholes RDP98 to RDP213 were drilled and most of them intersected significant disseminated tin mineralisation with only 4 holes out of 107 not intersecting anomalous tin values(<120ppm Sn).

The zone of anomalous tin values forms a roughly lenticular domical body, elongated NW-SE traceable for a distance of nearly 1km. The mineralisation is thickest along central axis and tapers off in all directions. The results of the Phase 3 and adjacent boreholes from other Phases however were sufficient to warrant calculation of the economic potential of the disseminated tin mineralisation in the area covered by the drilling. Previous underground mining within the disseminated mineralisation zone targeted the rich, tin-bearing pipes, which represent only a small proportion of the zone of the disseminated mineralisation. Thus, in areas with underground workings, a conservative figure of 5% by volume was discounted for the disseminated mineralisation which is investigated in this project.

Modelling and evaluation of the data gathered during previous drilling programmes has shown the presence of a body of disseminated tin mineralisation, 850m long, 250m wide and with a maximum thickness of 30m, a minimum thickness of 2m and an average thickness of 14m using a 200ppm cut-off grade. The average grade is 813 ppm Sn and this can be upgraded using a higher cut-off grade. Grades were considered to be sub-economic at the time of the exploration programme in 1978 and Tin price was at US$1,625/t (Clementson, 1979). The increase in global tin price, which now stands at about US$15,000/ton Sn (October 2009) makes this deposit highly attractive and it could be economically viable. The ore body could easily be mined by open cast methods. In this report, a cut-off grade of 200ppm Sn (0.02%) is used for resource calculation since it results in the best grade continuity of mineralisation. Using a maximum overburden thickness of 30m and cut-off grade of 200ppm Sn, the polygonal estimate for the anomalous disseminated tin-bearing body within the Lease Granite constitutes an indicated resource of 6.26 million tonnes with an average grade of 813.05 ppm Sn. The resource would yield 5,085 tonnes of tin. The kriged block estimate for the disseminated tin-bearing body constitutes an indicated resource of 6.06 million tonnes with an average grade of 528.69 ppm Sn and contains 3,203.86 tonnes, for a cut-off grade of 200ppm and 30m overburden limit.

The ore body is generally exposed at the surface with the dome crest having been removed by erosion and with very little overburden. In this study, maximum overburden thickness in the anomalous zone is 30m and is localised in the south-western edge of the area. The mineralisation is open ended to the northwest and also to the south-east .Further investigation is recommended in these areas. Small but very high grade tin pipes occur in the area and they could not be investigated by normal exploration methods due to their small size and unpredictability but they kept mining operations going for more than a decade on a small scale in the old Groenfontein mine in the seventies (Clementson, 1979). An opencast operation could expose the pipes which would enhance the average grade derived from the disseminated mineralisation which envelopes them. The financial valuation using polygonal estimates has yielded positive results suggesting that selective mining could be economically viable. The financial valuation using geostatistical estimates has yielded negative results suggesting that bulk mining may not be economically viable and should be discouraged based on the economic modifiers used.

This research is part of a bigger project by VM Investment Company aimed at re-evaluating the tin resource on two farms since the price of tin has gone up during the past decade. More investigation should be done to ascertain the economic factors in the conversion of the resource to reserve.

ACKNOWLEDGEMENTS

I would like to thank VM Investment Company for providing the dataset and a computer equipped with MapInfo/Discover, MapSource and other geological software and my supervisors; Prof Richard Minnitt and Prof Morris Viljoen for their invaluable advice and guidance throughout the research.

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CHAPTER 1: BACKGROUND AND OBJECTIVES OF RESEARCH

1.1 Summary and research questions

The evaluation of the tin resource on the farms Roodepoort 222KR and Groenfontein 227KR is based on a major drilling program that was carried out by Rand Mines in 1978 with the aim of investigating the economic viability of the remaining resource on the farms. In 1979, Clementson conducted a study of tin mineralisation on these farms and concluded that a potentially economic concentration of disseminated tin existed and could warrant further investigation. Clementson used traditional methods to evaluate the deposit such as manual drawings of sections and polygonal methods to establish the size and grade of the tin resource. The aim of this research is to carry out a geological, geostatistical and financial analysis of the remaining tin mineralisation by making use of modern day technology to re-model and assess its economic potential. The same data collected by Rand Mines in 1978 was captured onto excel spreadsheets, imported into MapInfo and then processed to establish a GIS database and produce geological and geostatistical models. Financial modelling was carried out using excel Discounted Cash Flow (DCF) analysis. In addition, comparable cost/market approach and option pricing techniques maybe used to evaluate the economic viability of the tin resource in question.

1.2 Rationale

The previous evaluation work was carried out at a time when tin price was relatively low, less than US$2,000/t (Clementson, 1979). The current tin price of about US$15,000/t (July 2009) makes this resource attractive and the re-evaluation is aimed at determining its economic viability. In the previous study, the areas that are covered by unmineralised hanging wall were not considered. It now seems probable that mineralisation below an overburden cover of up to 30m could be economically viable and add a significant amount to the overall resource tonnage. The use of modern-day technology such as MapInfo and Excel spreadsheets, not only provides a faster and more efficient means to process the data, but can also be used as efficient and effective databases for data storage and retrieval systems. The computer based methods produce quick results and can be easily manipulated for scenario analysis and re-evaluation should certain parameters change.

1.3 Aim and objectives

The objective of the study is to make use of modern-day technology to remodel and evaluate the economic potential of the tin resource on farms Roodepoort and Groenfontein using previously acquired data from earlier exploration programmes by Rand Mines during the late 70s. The available software are MapInfo for GIS database storage and manipulation, geological and geostatistical assessment and Excel for financial assessment. The models to be produced are geological, geostatistical and financial models. The financial model will be part of the feasibility study that will be used to assess and conclude the economic potential of the tin resource in question.

1.4 Methodology

The original borehole locations were in Cape Coordinate System LO29 and these coordinates were converted to UTM WGS84 Zone 35 South, using MapSource software and imported into MapInfo software. Old maps were scanned and geo-referenced in MapInfo for data manipulation. Initial modelling of the tin resource was carried out through hand contouring on paper and then digitising using AutoCAD software.

The digital terrain model was produced in MapInfo by making use of available borehole collar survey information and satellite imagery from Google Earth which was used to map the extent of the surface mineralisation with its distinctive spectral signature. The borehole collar information was imported into MapInfo and processed to produce a digital terrain model (DTM) which was them used to create a surface for the cross and longitudinal sections.

The geological model, cross and longitudinal sections were produced in MapInfo/Discover software. The available lithological and mineralisation information was imported into MapInfo/Discover software and processed to produce geological and mineralisation models.

The geostatistical models were produced in MapInfo/Discover software. The assay information was imported into MapInfo/Discover software and processed to produce suitable variogram models and kriged block models from which a mineral resource statement was made.

The financial model was manipulated using specifically designed models in Excel. Information on capital, production and operational expenditure was based on previously acquired data from local tin mines and other open pit operations in the country. NPV and IRR was the basis for concluding the economic viability of the tin resource in question.

1.5 Literature review

1.5.1 Locality and access

The properties of Roodepoort 222KR and Groenfontein 227KR are located on the Northern Limb of the Bushveld Complex within part of the Zaaiplaats Tin Mining Area about 30km north-west of Mokopane (Potgietersrus), as shown in Fig 1.1 and Fig 1.2. The property resides under Bosveld Magistrial authority. Access to the property is good and a proper infrastructure is in place. Electrical power is available from the national supplier, Eskom (Clementson, 1979).

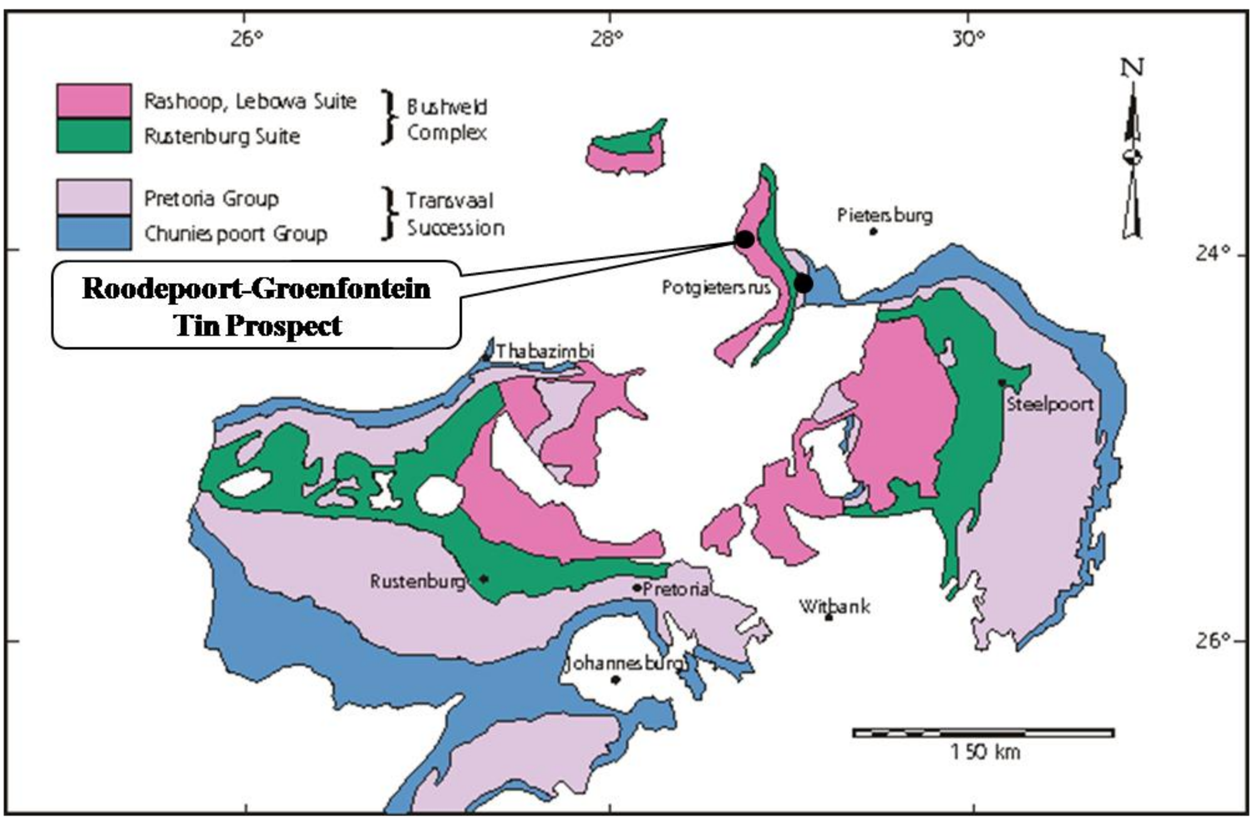

Fig 1.1: Location of Roodepoort-Groenfontein Tin Project on the acid phase of the Bushveld Complex on the Northern (Potgietersrus) Limb, (after Muindisi, 2004)

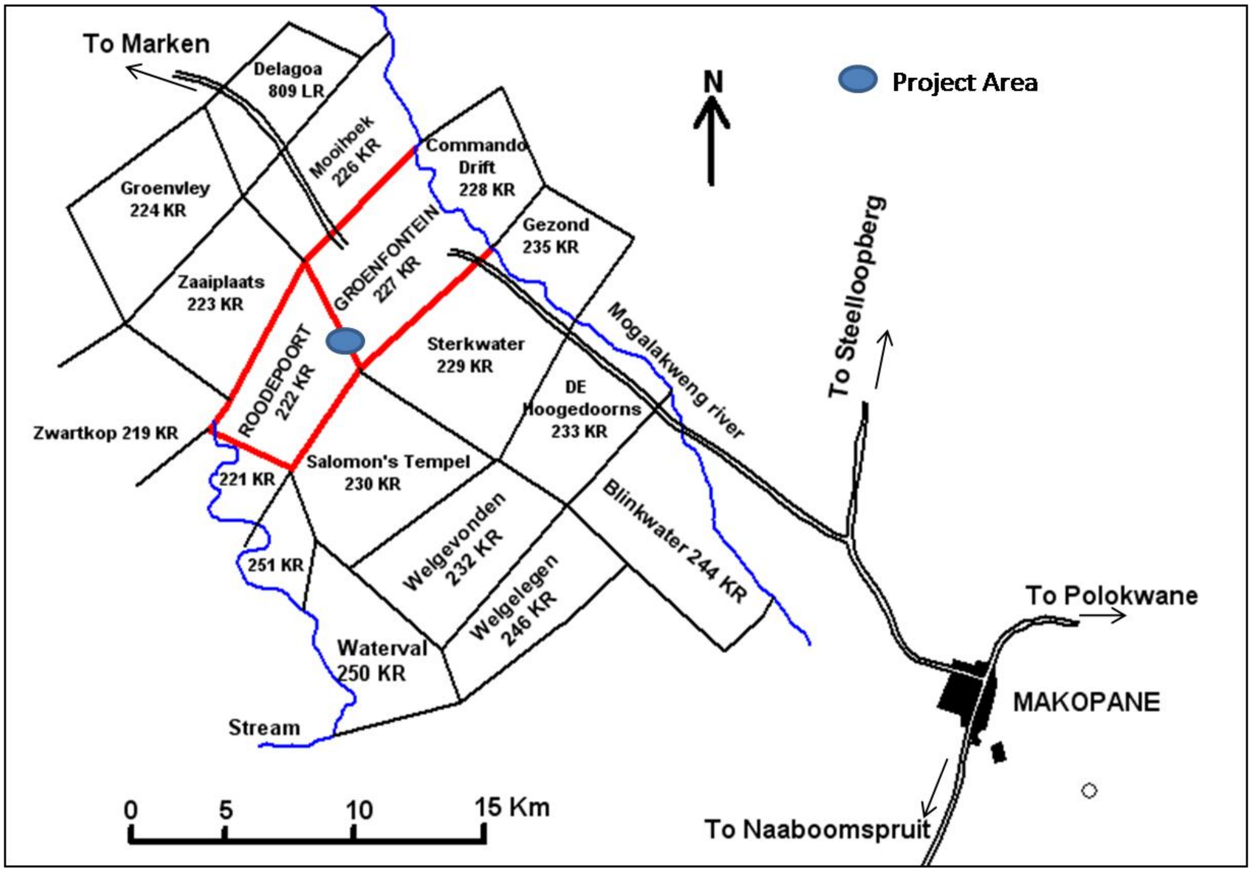

Fig 1.2: Locality Map of Roodepoort-Groenfontein Farms, (after Clementson, 1979)

1.5.2 History of Ownership

In 1907 Groenfontein Tin Mine (the majority of the workings are on Roodepoort) was opened by Transvaal Consolidated Lands mining company (TCL) and worked by the company until 1930. It was placed on a care and maintenance basis until 1939, after which a tribute to mine tin was granted to Mr A. Gilbertson. The tributor worked the high grade, pipe-like ore bodies in the lease granite and did some work on the recovery of eluvial tin in the Giant Quarry area, (Fig 1.3). The tribute did not produce much tin, about 4 tons a month of about 70% tin concentrate until the 1960s (Clementson, 1979).

On Zaaiplaats, mining commenced in 1908 by the Zaaiplaats Tin Mining Company (ZTM) and work was restricted to the farm Zaaiplaats until 1953 when ZTM was granted a tribute to mine tin ore from certain areas of the farm Roodepoort including areas of low grade disseminated cassiterite within the Bobbejaankop Granite around the No XIII Quarry and from Bobbejaankop Hill itself (Fig 1.3) and this agreement, with some alteration, was still in effect in 1979.

In 1979 ZTM was still recovering appreciable amounts of tin from the Roodepoort tribute, (Clementson, 1979.) In the 1960s the Gilbertson tribute on Groenfontein was transferred to Zaaiplaats Tin Mining Company. In 1979, the ZTM tribute was still in effect and the company was working tin pipes in the Lease Granite on a small scale and also hand sorting old dumps to recover high grade ore. In 2001 Quickstep 169(Pty) was issued with mining permit (No 5/2001) to mine on the farm Zaaiplaats 223KR.

1.5.3 General geology and tin mineralisation

Tin mineralisation is confined to the acid granitic phase (the Rashoop or Lebowa Suite) of the Bushveld Complex (Fig 1.1). The granitic phase consists of a +/- 2.8km thick stratiform granite (the Nebo granite) which includes stocks forming upper mantos of volatile enriched red granite which is sometimes tin-mineralised as for example the Bobbejaankop Granite of the Zaaiplaats area (Fig 1.3). Tin deposits related to the Bobbejaankop Granite occur as endogenic pipes and disseminations or as exogenic fissure veins, fault breccias and replacement bodies (Lee, 1977). The Bobbejaankop Granite at Zaaiplaats is intrusive below a thick sheet of granophyres (the Main Granite) which acted as a volatile trap. In general, the potential tin bearing areas are located high in the granite succession and require a roof (e.g. granophyres, sediment, or early granite) to act as a volatile trap (Lee, 1977). The Lease Granite is regarded as a lithological variation of the Bobbejaankop Granite (Clementson, 1979). Economic tin mineralisation has only been found in both Lease and Bobbejaankop Granites. The mineralisation on farms Groenfontein and Roodepoort is in a fine grained Lease Granite which is the focus area for the present study.

1.5.4 Local geology and tin mineralisation

The geology of the farms is dominated by the acid phase of the northern limb of the Bushveld Complex. Generally, the Main Granite in the area (Nebo granite) was intruded by the more volatile-rich Bobbejaankop Granite which occurs as dome like plutons along the upper margin. The Bobbejaankop Granite is a coarse grained, red, biotite and potassic-feldspar bearing granite which is overlain by a fine grained leucocratic Lease Granite. The Lease Granite is immediately overlain by a quartz-feldspar Pegmatite Zone, (Fig 1.3). The Lease Granite and the pegmatite dip at 10^o-20^o to the south west.

The Lease Granite forms a thin lenticular, concordant sheet between the Bobbejaankop and the Main Granites (Fig 1.3), with a maximum thickness of 120m, (Clementson, 1979). Tin mineralisation is restricted to the Lease and Bobbejaankop Granites and it occurs in pipe-like bodies, sub-horizontal lenticular bodies and as sub-horizontal disseminated low grade bodies within both granites. All tin mineralisation is in the form of cassiterite (SnO_2). Clementson (1979) gave a description of the mineralisation styles which is summarised below:

- **Pipe- like bodies**: these are prominent in the Lease Granite but have also been reported to occur in the Bobbejaankop granite e.g. on Zaaiplaats 223KR. The cassiterite concentration is up to 70%+ with an average of between 12% and 30%. The pipe-like bodies are roughly circular in cross-section with diameters varying from a few centimeters up to 12m and lengths from a few metres up to 1200 metres. The attitude varies from horizontal to vertical and they can be barren.
- **Lenticular Ore-Bodies**: These occur in the Lease Granite immediately below the pegmatite zone and represent “bubbles” of tin bearing fluids which were trapped beneath the impermeable pegmatite.
- **Disseminated Ore Bodies**: Bobbejaankop and Lease granite have extensive areas of disseminated cassiterite within them with grades ranging from 0.02% to 0.6%, averaging 0.08%. The deposits are high tonnage and potentially bulk mineable.
- **Eluvial and Alluvial Deposits**: These surficial deposits have been worked in the past and further economically exploitable deposits could be located.

1.5.5 Previous South African tin production and markets

Prior to its closure in 1979, Zaaiplaats Tin Mine (Quickstep 169) was the sole producer of tin in South Africa and 80% of its production was consumed by South Africa’s largest metal company, ISCOR and the rest by other local steel manufacturers (Zelda Real Estates, 2006). The export market was not exploited due to local demand. Tin concentrate, in the order of 55 % tin was sold FOT (Free-On-Trucks), ex plant and transported by purchasers..

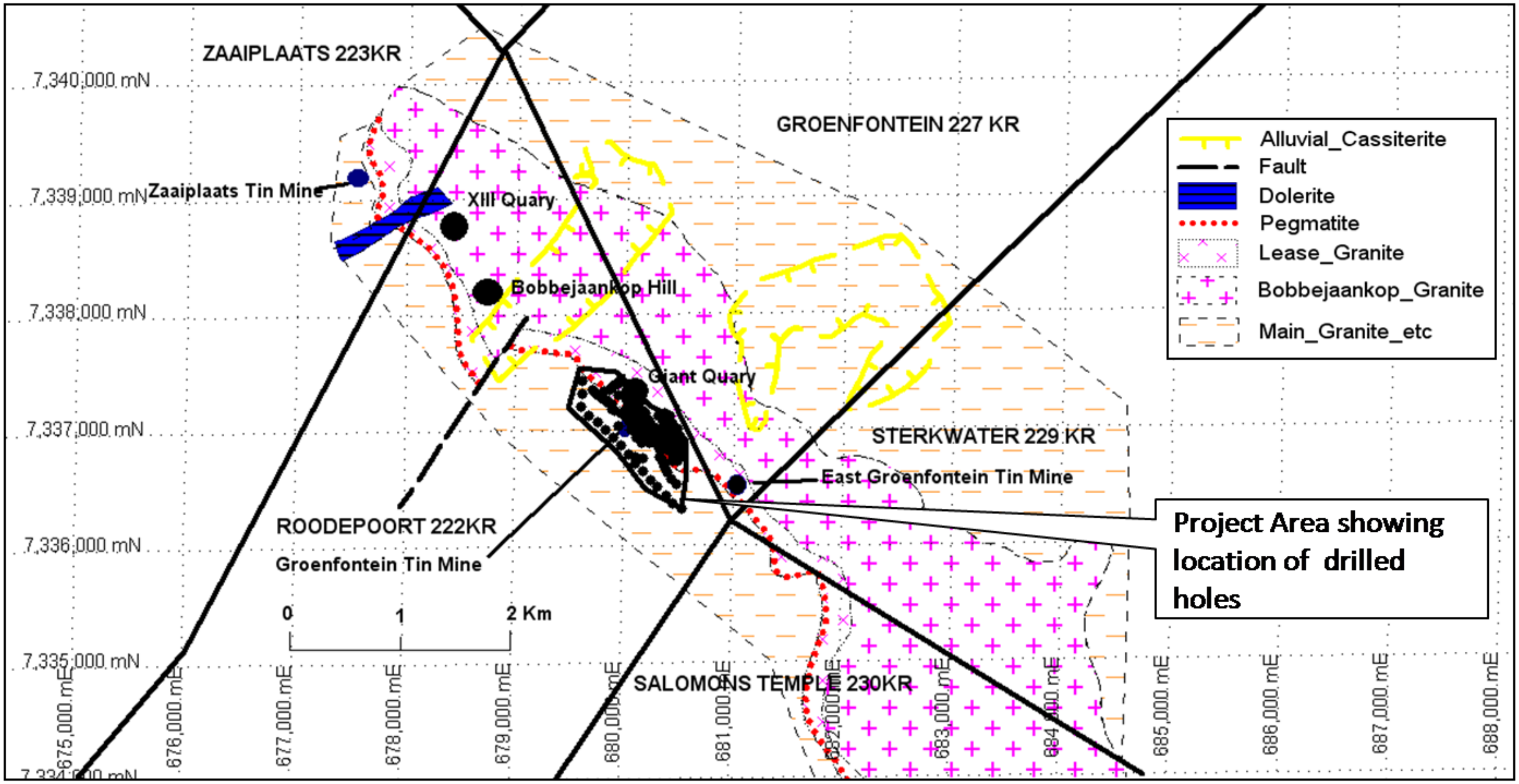

Fig 1.3: Geology and mineralisation on the farms Roodepoort 222KR and Groenfontein 227KR , (after Cain, 1977)

1.6 Project background and exploration history

Tin was discovered in the area in 1906 and has been commercially mined since 1907 by Zaaiplaats Tin Mining Company from Zaaiplaats 223KR, Roodepoort 222KR and Groenfontein 227KR; the latter two have been worked under tribute from TCL (Clementson, 1979) A report for Rand Mines entitled "Tin Potentialities on TCL Farms", 1962, by Mr A Kriek summarised the value and importance of tin as a target for exploration and outlined the potential of several TCL properties in South Africa. Welgevonden-Welgelegen and Roodepoort-Groenfontein, (Fig 1.2) within the Potgietersrus Tin Field were identified as being prime targets for intensive exploration (Clementson, 1979).

1.6.1 Styles of mineralisation and their economic potential

1.6.1.1 Pipe like ore-bodies

The pipe like ore-bodies in Lease Granite or Bobbejaankop Granite were not considered to be a major source of ore for a large scale operation. This was due to their small size, irregularity and unpredictable nature.

1.6.1.2 Lenticular ore bodies in Lease Granite

There are lenticular ore bodies in Lease Granite. These bodies could not be of prime target for exploration because of their unpredictable nature.

1.6.1.3 Disseminated cassiterite

The disseminated zones were considered to be the most attractive exploration targets because of their potential for large volumes of predictably mineralised granite. This fact was enhanced by the possibility of lenticular ore bodies being associated with areas of disseminated cassiterite within the Lease Granite. In addition, known disseminated cassiterite ore bodies tend to form a predictable 2-10m thick zone concordant with the pegmatites and are usually less than 30m below the pegmatites. The strike length of known mineralised Lease Granite on surface straddles the Roodepoort-Groenfontein boundary and exceeds 600m with the extension of the ore down dip projected to be in excess of 150m (Clementson, 1979). Zaaiplaats worked disseminated cassiterite in places such as on Roodepoort, on Bobbejaankop Hill itself and along the northern portion of Roodepoort-Zaaiplaats boundary shown in Fig 1.3 (Clementson, 1979).

1.6.1.4 Alluvial deposits

Two alluvial areas, to the north and north east of Groenfontein Tin Mine were regarded as possible targets for workable alluvial tin deposits (Fig 1.3). It was concluded that the favourable modes of mineralisation for investigation were; the disseminated cassiterite bodies and associated lenticular ore bodies within the Lease Granite and disseminated cassiterite in the Bobbejaankop Granite and two alluvial tin deposits. The conclusions formed the basis for subsequent exploration programmes in the area.

1.6.2 Exploration follow-up

1.6.2.1 Investigation of disseminated cassiterite in Lease Granite:

In 1963, twelve holes RDP001-012 which were drilled randomly gave a roughly correlatable zone of anomalous tin mineralisation. The mineralisation pattern was crude due to large inter-hole spacing (Fig 1.4).

1.6.2.2 Investigation of two alluvial tin deposits: 1975-6 drilling programme

In 1975-6 in an area to the north east of Groenfontein Tin Mine, 66 percussion holes were drilled in an area with favourable drainage. In Groenfontein Valley, a further 23 holes were drilled, however, the information on these boreholes could not be found at the time of compiling this report. The holes were drilled to below bedrock and sampled continuously at every 50cm. The results indicated that there were no well-defined layers of tin mineralisation with low tenor of cassiterite content making alluvial tin an unattractive proposition (Clementson, 1979).

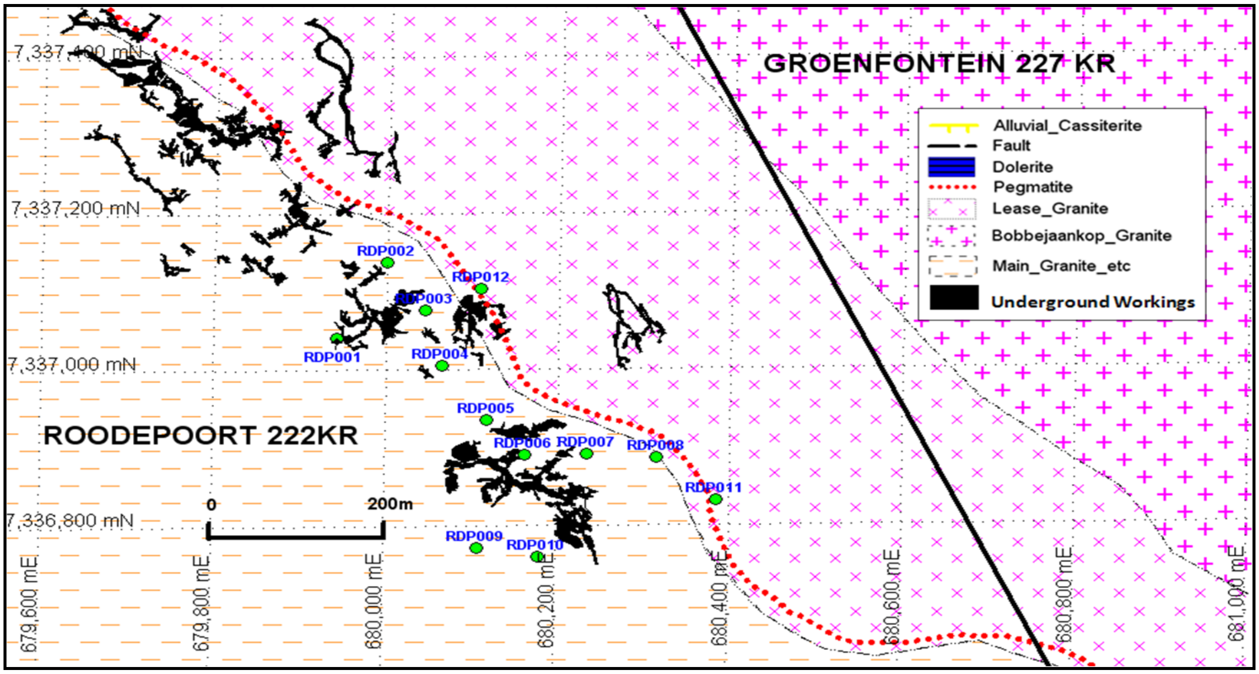

Fig 1.4: Drilling programme of 1963 showing distribution of RDP001 to RDP012.

1.6.2.3 Investigation of disseminated cassiterite in the Lease and Bobbejaankop Granite

In 1976, a detailed systematic sampling over Groenfontein-Roodepoort was carried out. Granite chip samples and soil/alluvial/eluvial samples were collected, initially on a 50m by 50m grid and later on a 10m by 5m grid on some of the more interesting areas. The results of the 50m by 50m grid sampling only revealed known tin occurrences, (Fig 1.3), as follows:

• Giant Quarry (Lease Granite)

• No. XIII Quarry (Bobbejaankop Granite)

• SE Bobbejaankop Hill (Bobbejaankop Granite)

• East Groenfontein Mine (Lease Granite)

The approximate location of these areas is shown in Fig 1.3. Two trends of tin mineralisation were established:

- **Zaaiplaats Trend,** within the Bobbejaankop Granite running NW-SE with the XIII Quarry being regarded as the partly eroded SW limb of the Zaaiplaats trend (Fig 1.5).
- **Groenfontein Trend,** within the Lease Granite which trends N-S (Fig 1.5) and appears to be spatially related to major pipe systems occurring on Groenfontein No 9, No 22 and

Gilbertson Pipes. The results of a 5m by 10m grid sampling over the Giant Quarry revealed two well defined trends of low grade tin mineralisation displaying a marked spatial relationship to the No. 9 and No. 22 pipes (Clementson, 1979).

It was deduced that the Lease Granite trends could extend south, parallel or sub-parallel to the Lease–Main Granite contact, beneath the barren Main Granite cover and could hold considerable potential for large tonnage low grade ore and the idea was supported by the 1963 drilling programme (RDP 1-12) as shown in Fig 1.4. The southerly extension (the Groenfontein trend) of the low grade surface mineralisation (Fig 1.5) was therefore, the target for a subsequent drilling programme.

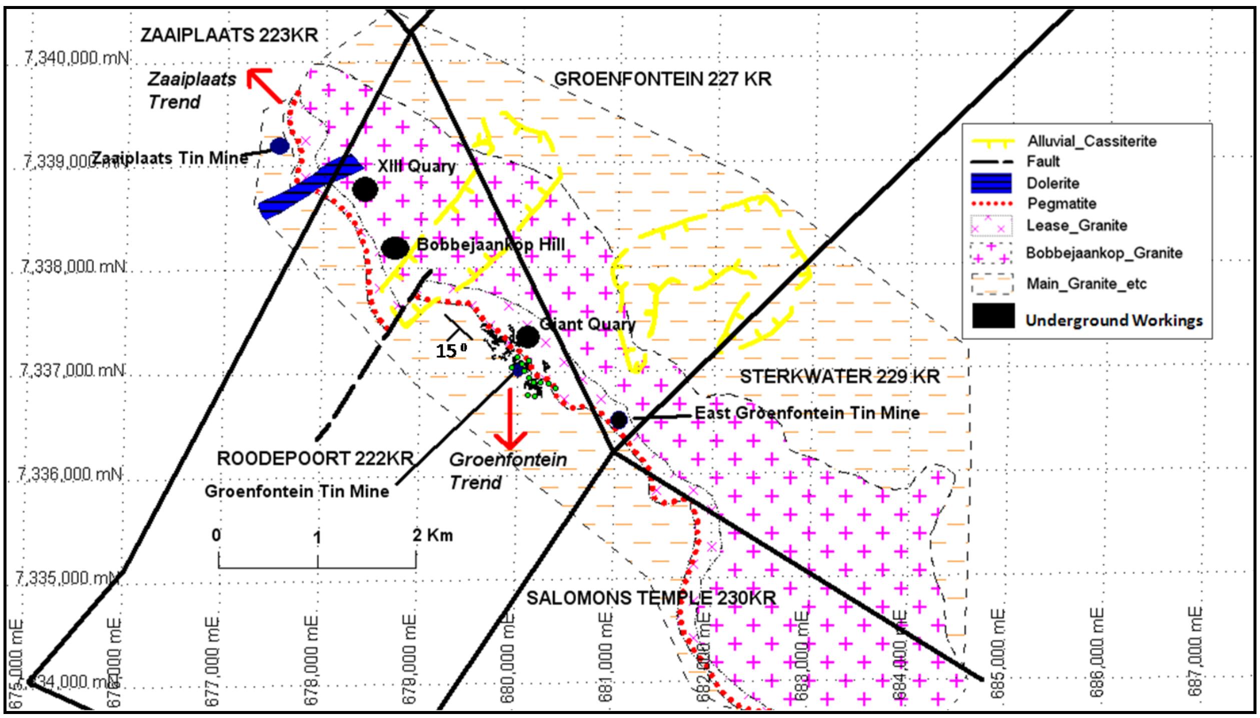

Fig 1.5: Mineralisation localities and the proposed Groenfontein and Zaaiplaats mineralisation trends

1.6.2.4 Investigation of the southern extension of the tabular ore body

In 1978, a major drilling program was carried out on the tabular mineralisation occurring in the Lease granite. The aim was to investigate the economic viability of the southern, down dip extension of the tabular tin mineralisation on Roodepoort and Groenfontein. A series of diamond holes alternating with percussion drill holes (RDP13 to 53) were drilled along a NW-SE line, roughly parallel to the Main-Lease Granite contact but down dip (Fig 1.6).

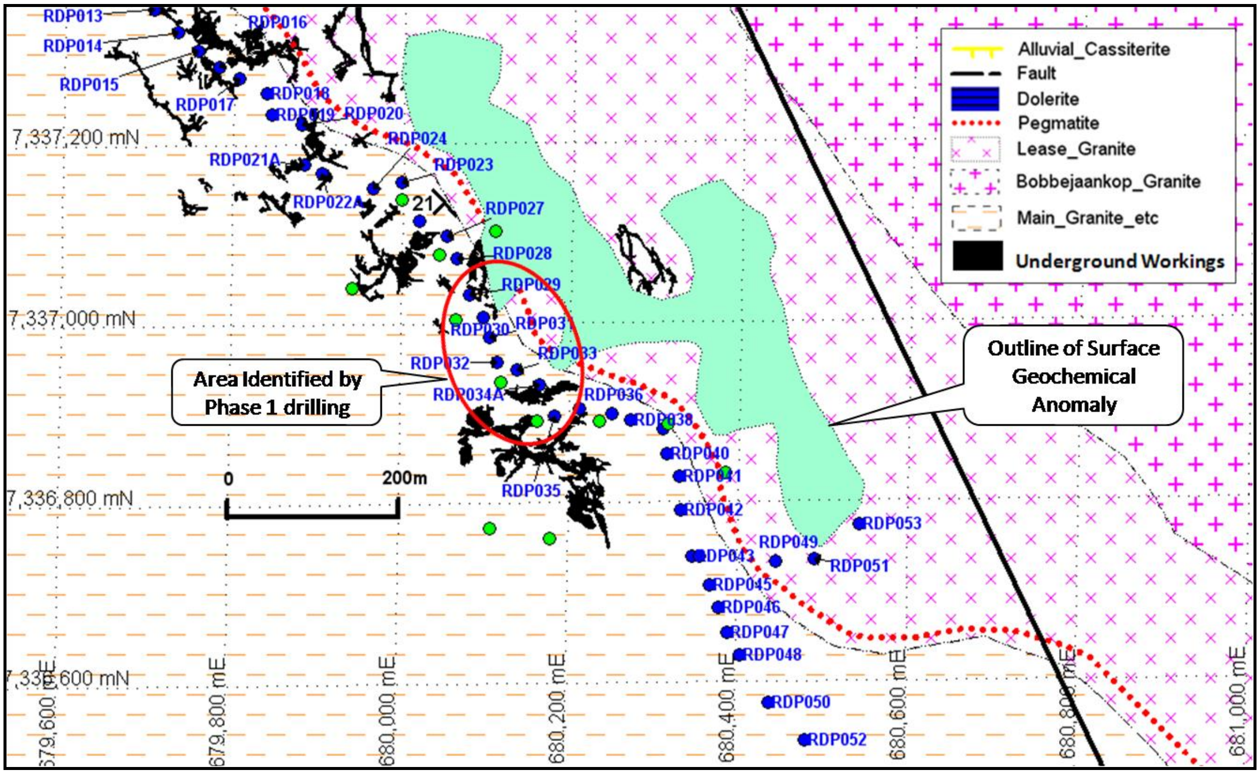

Fig 1.6: Phase 1, RDP013 to RDP053 drilled down-dip and parallel to the Main-Lease granite contact.

The holes were placed at 30m intervals and BX sized core was recovered. The diamond holes were drilled to about 40m below the Lease-Main contact as defined by the base of the pegmatite zone. Previous knowledge revealed that mineralisation is restricted to about 30m of the top of the Lease Granite. The core was logged and sampled from the top of the pegmatite to the end of the hole. Sampling involved splitting the core in half and sampling continuously over 1m intervals, thus, all the pegmatite and the intersected portion of Lease Granite were sampled. The unsampled halves were stored in old buildings at Groenfontein Tin Mine.

Samples were analyzed for Sn, Cu and CaF_2 but later for Sn only. There was no correlation between Sn and Cu and also between Sn and CaF_2. Cu values in pegmatite and Lease granite were low, averaging 44ppm Cu. The Lease Granite contains CaF_2. The maximum recorded value was 4.4% CaF_2 as shown in Table 1.1 (Clementson, 1979).

Intermediate 4.5 inch percussion drill holes were sampled by a cyclone system or an enclosed system of catch trays over every metre drilled. The sample material was split using riffle splitters and one half submitted for analysis while the other half was stored in the labelled bags at Groenfontein Tin Mine. The holes were drilled to about 40m below the calculated depth of the top of the Lease Granite (Clementson, 1979). The distribution of boreholes is shown in Fig 1.6.

Boreholes RDP049, 51 and 53 were drilled off the original NW-SE line (Fig 1.6), to the east of the geochemical anomaly to the depths of 60m and were collared in Lease Granite. This was done to check if the mineralisation persisted at depth to the east.

RDP047 was drilled to a depth of 72m below the Main-Lease contact to check the assumption that mineralisation was restricted to the upper 30m of the Lease Granite. RDP025 was not drilled as it fell adjacent to RDP002 (Figs 1.4). Several holes which were sited over underground workings were repositioned as close to the original site as possible.

A second line of holes (RDP054 to 67), 200m down dip, SW of the first line were planned to intersect the projected ore body further down dip from Phase 1 holes (Fig 1.7). The holes were spaced at 100m intervals, each hole piloted by percussion drilling to about 20m above the Main-Lease contact, as calculated from contact intersections in Phase 1 diamond holes, then diamond drilled to about 40m below contact. Logging and sampling procedure was the same as in Phase 1.

Table 1.1: Tabulation of copper and fluoride results

Hole ID	Width Sampled(m)	Copper(Cu)				Fluorite(CaF_2)			
		Max(ppm)	Min(ppm)	Cumulative(ppm)	Average(ppm)	Max(%)	Min(%)	Cumulative(%)	Average(%)
RDP14	41	363	20	1806	44	2.43	0.22	53.78	1.31
RDP16	43	335	14	3127	73	1.88	0.25	53.48	1.24
RDP18	45	23	6	538	12	1.52	0.66	51.25	1.14
RDP20	42	216	8	3171	76	3.15	1	72.51	1.73
RDP24	42	133	9	1001	24	4.4	1	70.17	1.67
RDP42	47	94	15	1751	37	2.09	0.12	46.08	0.98
RDP44	-	-	-	-	-	2.56	0.1	64.09	1.21
RDP46	54	-	-	-	-	1.33	0.08	43.98	0.81

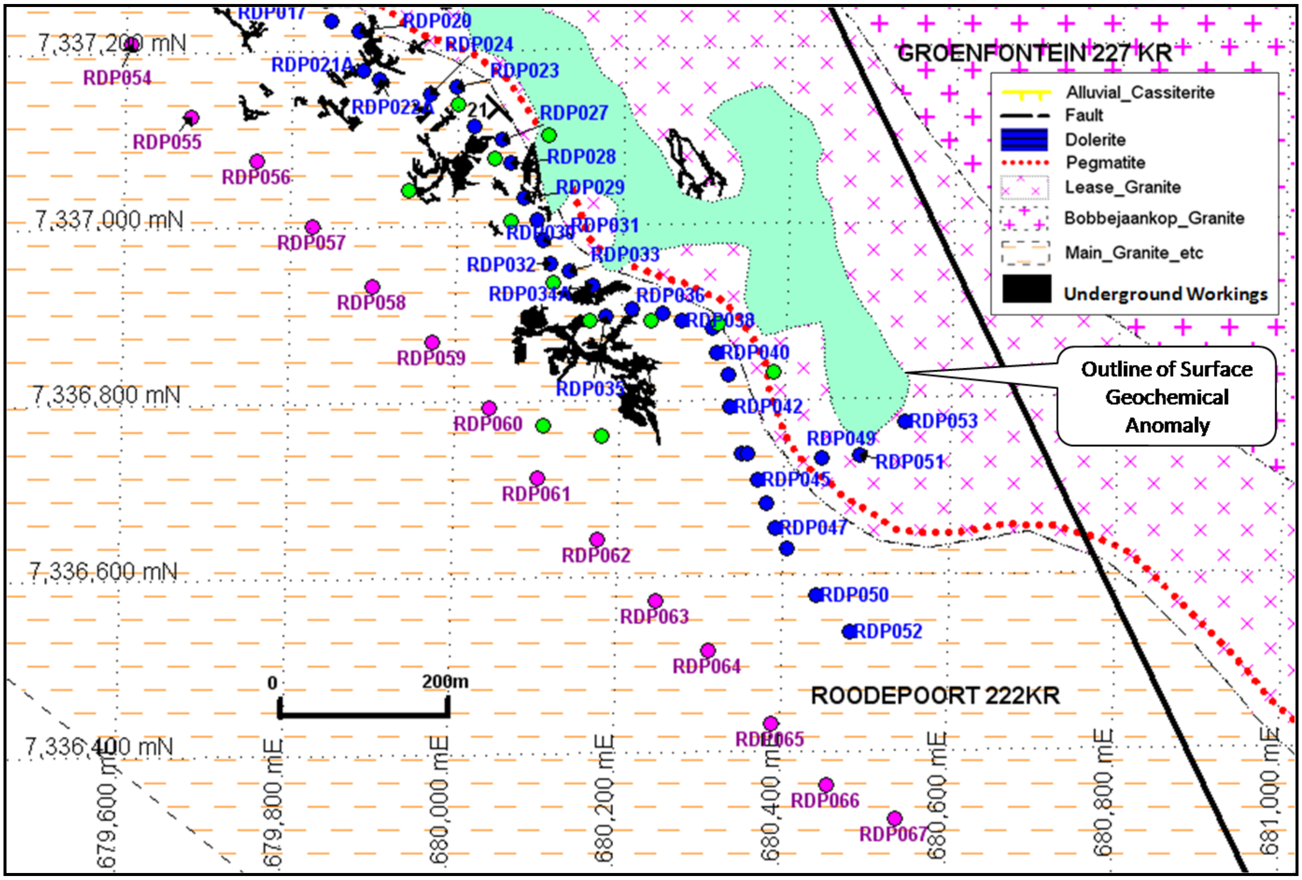

Fig 1.7: Phase 2, RDP054 to RDP067 drilled 200m down-dip from Phase 1 RDP013-053 line.

The information gathered about the lithologies in the area is the same as that previously known. The Main-Lease contact is fairly sharp with an average dip of 21^{o} to the SW.

Phase 2 holes were all barren and have shown that any mineralisation in the Lease Granite is restricted to the surface outcrop area and only extends for a very limited distance down dip, refer also to Fig 2.2 and Fig 3.2. Phase 1, RDP 13-52, excluding 49 and 51 gave interesting results and a correlation of anomalous values is possible but values were low except RDP 29 to 35 values, (Appendix 1). Clementson (1979) noted that RDP 16, 18 and 20 had good values but the grades could be due to the association with Pegmatite, which is generally too unpredictable to be of economic significance.

Boreholes RDP 23, 42, 43, 45 and 48 have some good values but these are isolated and flanked by barren holes. Such occurrences could be adjacent to tin bearing pipes in the Lease Granite which are generally too unpredictable to be of economic importance. Boreholes RDP 29, 30, 31, 32, 33 and 35 have good values and this is the only zone discovered by Phase 1 and 2 to be of any possible economic potential (Fig 1.7).

Towards the completion of Phase 2 drilling, RDP068 to RDP074 were drilled at 30 to 50m intervals, (Fig 1.8) to check whether surface geochemical anomalies presumed to extend SSW had not been misinterpreted and could actually trend to the NW in Lease or Bobbejaankop Granite. In addition RDP077-87 and RDP088-97 were percussion drilled to about 10m over the two geochemical anomalies where disseminated cassiterite is visible in Lease Granite (Fig 1.8). RDP86 was deepened to 166.2m in Bobbejaankop granite to check for possible mineralisation.

Phase 2 proved that the mineralised zone does not extend down dip and thus, any extension of this zone must be up-dip towards the surface outcrop. This was eventually substantiated by Phase 3 drilling.

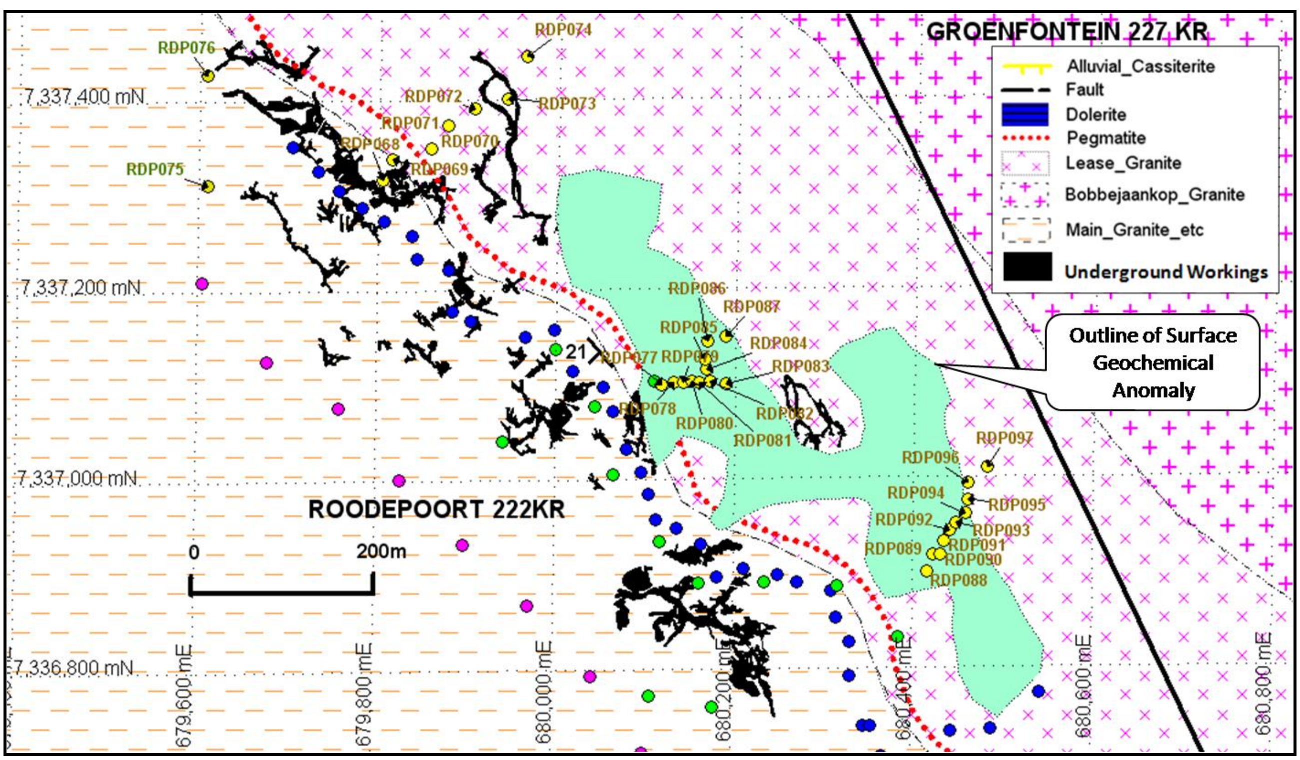

Fig 1.8: Phase 2 extension showing RDP068-074 and RDP077-097 drilled to check extent of surface geochemical anomaly, (RDP075 and RDP076 are inclined holes)

1.6.2.5 Investigation of the surface geochemical anomaly

The aim of phase 3 drilling was to investigate the extent of the surface outcrop proved to be of potentially economic significance by Phase 2 drilling. A grid of percussion drilling over the major portion of the surface geochemical anomaly (Fig 1.9), RDP98 to 213, was drilled on a 30m by 30m borehole spacing. The boreholes were collared in Lease Granite. The final depth of the holes was determined by extrapolating the known base of mineralisation in previously drilled holes over the grid and drilling down to that predictable base.

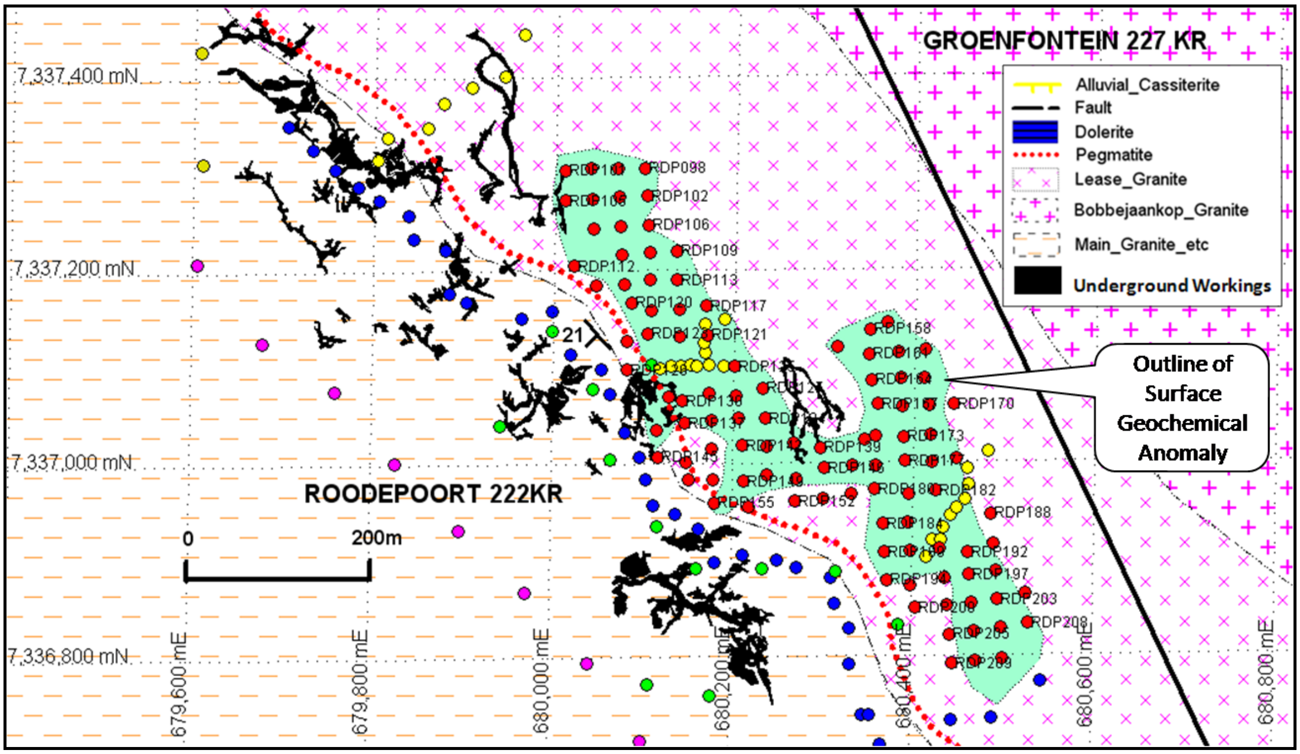

Fig 1.9: Phase 3 drilling, RDP098-213, over surface geochemical anomaly (drill holes shown in red).

Most of the boreholes RDP98 to 213 intersected significant tin mineralisation and only 4 holes out of 107 did not intersect anomalous tin values (<120ppm Sn). The sample results indicated that many of the holes had not penetrated the mineralised zone completely and had ended in mineralised granite. Analysis of results enabled estimates to be calculated for the 'undrilled' portion of the mineralisation and a reasonably clear picture of ore body was derived by Rand Mines. It was also observed that the zone of anomalous tin values forms a roughly lenticular body, elongated NW-SE and tapering off in all directions. The ore body maximum thickness could not be accurately determined as it was not fully penetrated in many of the holes. Accurate prediction of the form of the ore body was rather difficult since much of its upper portion has been removed by erosion. The results of the Phase 3 and adjacent boreholes from other Phases were sufficiently good to warrant calculation of the economic potential of the area covered by the drilling.

1.6.3 Mineral resource statement

The evaluation of the surface outcrop ore body was estimated from polygons and the conclusion reached is that the anomalous disseminated tin-bearing body within the Lease Granite on Roodepoort-Groenfontein comprises a measured resource of 2.9 million tons of ore averaging 817ppm Sn using a cut-off grade of 120ppm.

If a cut-off grade of 500ppm is taken, measured resource decreases to 1.7 million tons with an average grade of 1,196 ppm Sn. Further block selection could upgrade the resource. The indicated potential resource of the ore body was demonstrated to be 4.8 million tons of ore at an average grade of 688 ppm. A significant ore body was modelled on Roodepoort-Groenfontein which was considered to warrant serious consideration for opencast mining. Grades, however, were considered to be sub-economic at the time with a tin price of R12,998/t in March 1979 (Clementson, 1979).

It was pointed out that:

- careful mining selection and/or
- an increase in tin price may render the deposit economically viable.

1.6.4 Recommendations

It was noted that there are additional areas of known mineralisation within the Bobbejaankop Granite on Roodepoort at No. XIII Quarry and Bobbejaankop Hill, which have not been investigated since parts of these areas were being mined by Zaaiplaats. Thus, it was recommended that a further thorough investigation be carried out to delineate and assess any further possible resource on Roodepoort-Groenfontein, especially within the Bobbejaankop Granite. It was noted that an important factor to bear in mind in a viability study would be the occurrence of small but very high grade tin pipes known to occur in the area. Although the pipes could not be investigated by normal exploration methods, they have kept mining operations going for at least a decade on a small scale in the old Groenfontein mine. Thus, an opencast operation would expose the pipes and enhance the average proven grades (Clementson, 1979).

CHAPTER 2: RESOURCE MODELLING

2.0 Introduction

The present study uses information gathered in the 1978 Drilling Programme to re- evaluated the Roodepoort-Groenfontein tin mineralisation. The increase in global tin price, which now stands at US$15,000/ton (November 2009), more than seven times the price in March 1979 (R12,998/ton Sn or US$1,625, Clementson, 1979), makes this deposit highly attractive and the deposit could be economically viable.

The evaluation of the tin resource includes an analysis of the mineralisation thickness (m), tin grade (ppm) and tin content (m*ppm) and by means of hand contouring and geologically based interpretation of the data coupled with the construction of sections in MapInfo/Discover software. The area with historical underground mine workings has been included in this study as only the pipes were mined in this area and therefore potential must exist for further disseminated mineralisation. The extent of the old underground mine workings in relation to the zone of the disseminated mineralisation is shown in Figure 2.5 and is also well illustrated in Figs 3.1 to 3.5.

2.1 Contour modelling

2.1.1 Mineralisation thickness (m)

The disseminated mineralisation is thickest (15-30m) in the central portion of the project area where a zone extending for about 425m along strike by 50m in width is present (Fig 2.1). This forms the core of a more extensive zone of mineralisation, which, using a 5m thickness cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 2.1) where the mineralised zone has not been fully investigated. These areas could offer an additional tin resource, (Fig 2.1). The area of exposed mineralisation is demarcated from that overlain by unmineralised rock in Figure 2.1. A cut-off grade of 200 ppm Sn corresponds fairly closely to the 5m thickness of the zone of disseminated mineralisation.

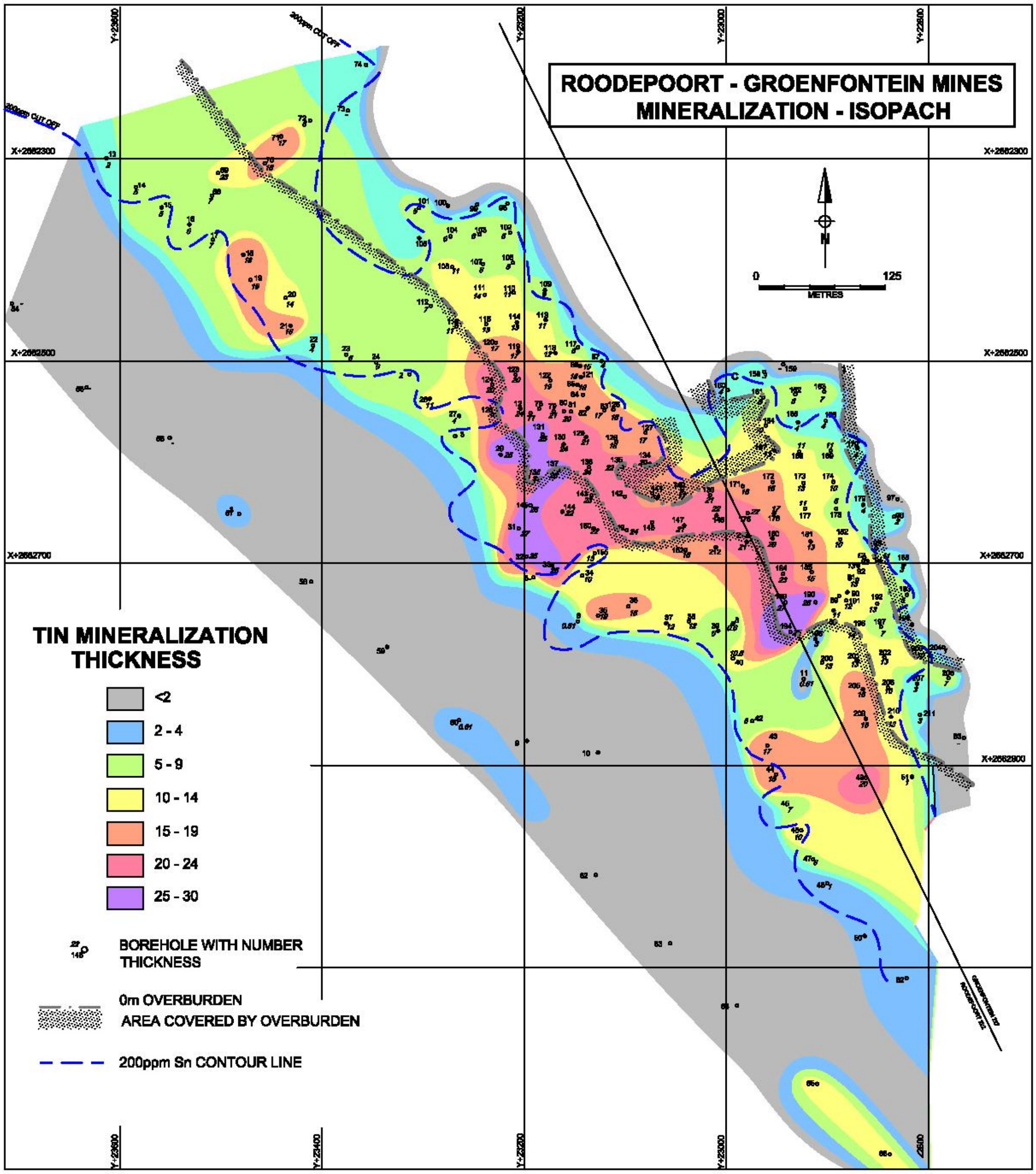

Fig 2.1: Mineralisation thickness (m) isopach map of the disseminated mineralisation zone

2.1.2 Grade (ppm Sn)

The mineralisation grade is highest (>1,000 ppm Sn) in two distinct zones, one in the central south-eastern and the other in the central north-western portion of the mineralisation manto, (Fig 2.2). The grade of greater than 500ppm Sn encompasses the higher grade areas and extends for about 600m along strike by 90m in width (Fig 2.2). This forms the core of a more extensive zone of mineralisation, which, using a 200ppm cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 2.2) where the mineralised zone has not been fully investigated. These areas could offer an additional tin resource (Fig 2.2). The area of exposed mineralisation is demarcated from that overlain by unmineralised rock in Figure 2.2.

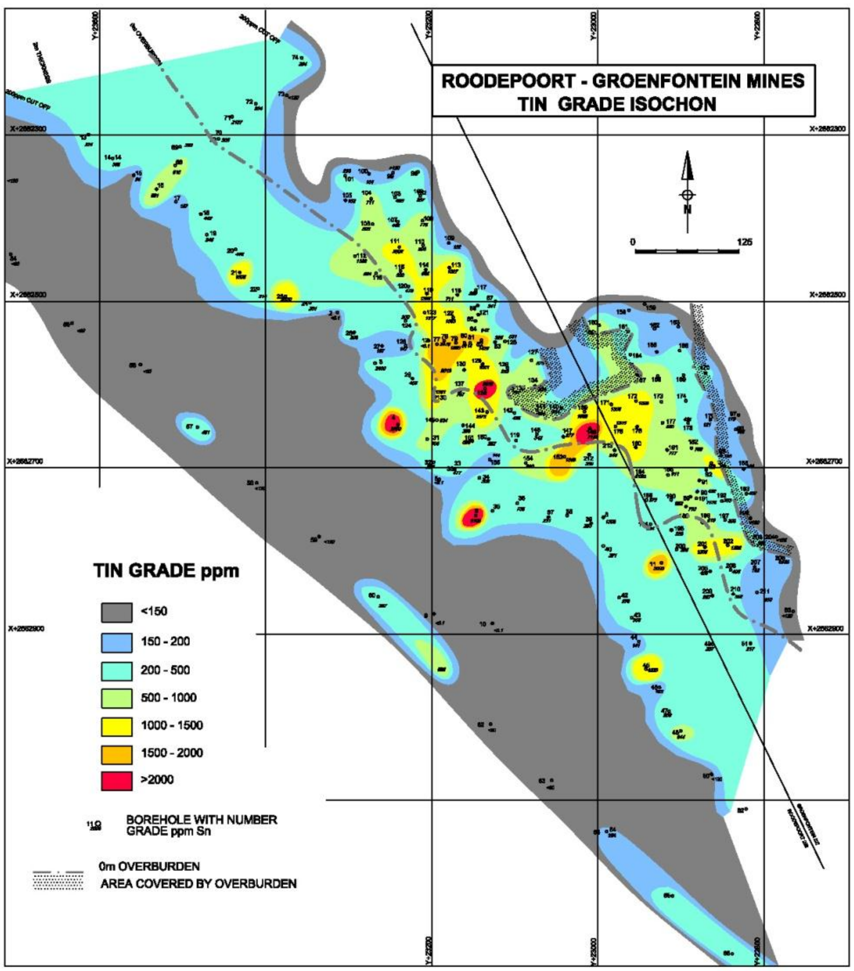

Fig 2.2: Tin grade(ppm) isochon map of the disseminated mineralised zone

2.1.3 Content (m*ppm Sn)

As with the observed grade distribution, two distinct zones of high tin content greater than 15,000m*ppm can be observed, one in the south-eastern and the other in the north-western portion of the investigation area (Fig 2.3). The content greater that 5,000m*ppm Sn extends for about 600m along strike by 90m in width (Fig 2.3). This forms the core of a more extensive zone of mineralisation, which, using a 2,000m*ppm cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 2.3) where the mineralised zone has not been fully investigated. These areas could offer an additional tin resource (Fig 2.3). The area of exposed mineralisation is demarcated from that overlain by unmineralised rock and a close coincident between 200ppm cut-off and 2,000m*ppm contours can be observed in Fig 2.3. These cut off grades also correspond fairly closely with the 5m thickness contour (Figs 2.3 and 2.4). This reveals a close relationship between thickness and grade, i.e. the thicker the mineralised zone the higher the grade.

2.1.4 Overburden Thickness (m)

The mineralised tin granite is largely exposed on the surface and takes the form of a broad anticlinal structure with a NW trend (Figures 2.4). Three limbs can be observed:

- the SW limb of the main mineralised zone with a general dip of about 10-15^{o} to the SW, which is the general dip of Bushveld rocks in the area
- the Eastern limb with a general dip of about 10-15^{o} to the NE
- the NNW limb with a NNW general dip direction

The overburden thickness increases down dip on all three limbs. The 30m overburden thickness is shown in relation to the 200ppm Sn cut-off contour (Fig 2.4). The 200ppm cut-off is at an average depth of 30m below overlying barren rocks on the SW and is mostly exposed on surface on the NE side of the domical structure.

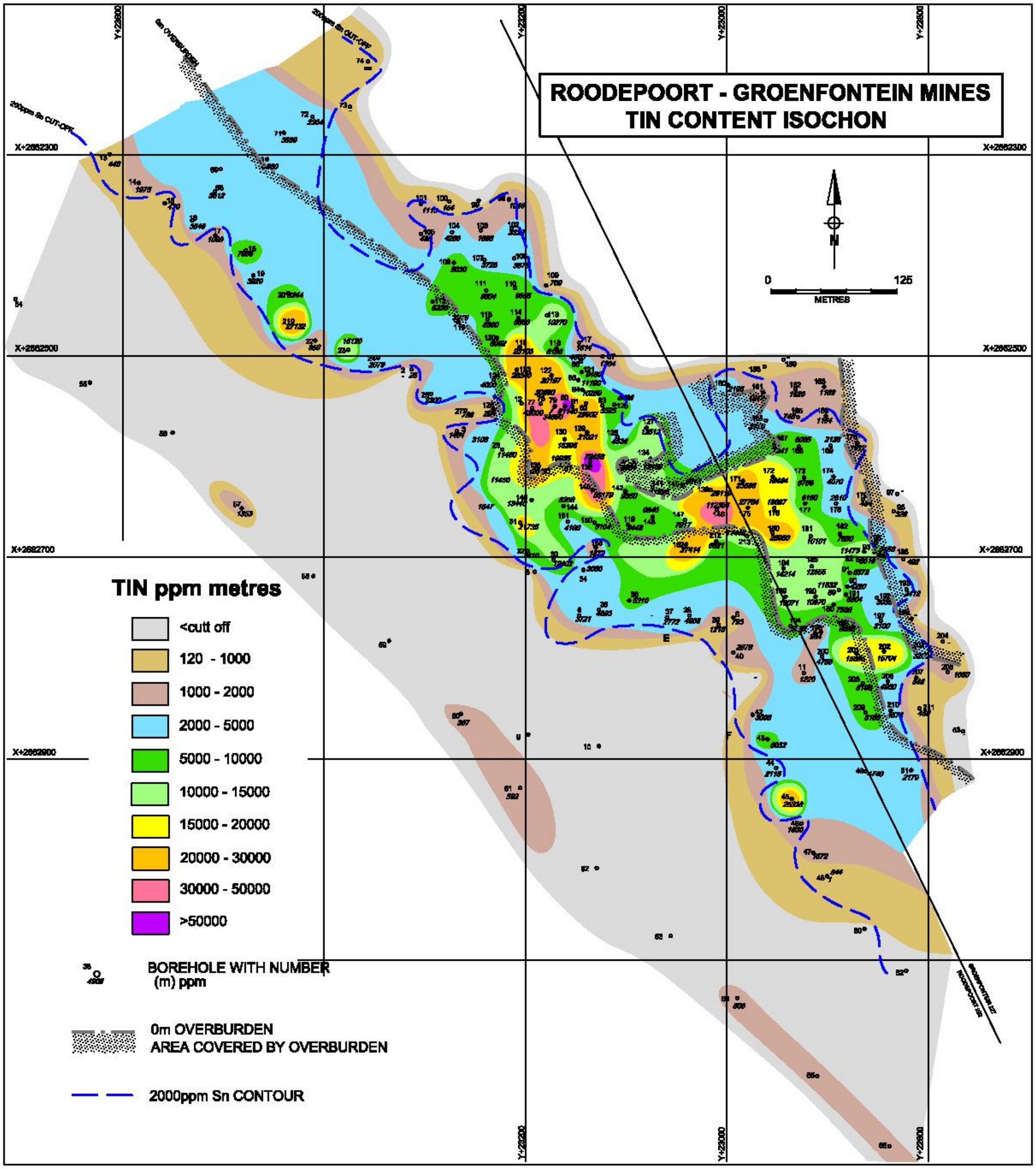

Fig 2.3: Tin content (ppm) isochon map of the disseminated mineralisation zone

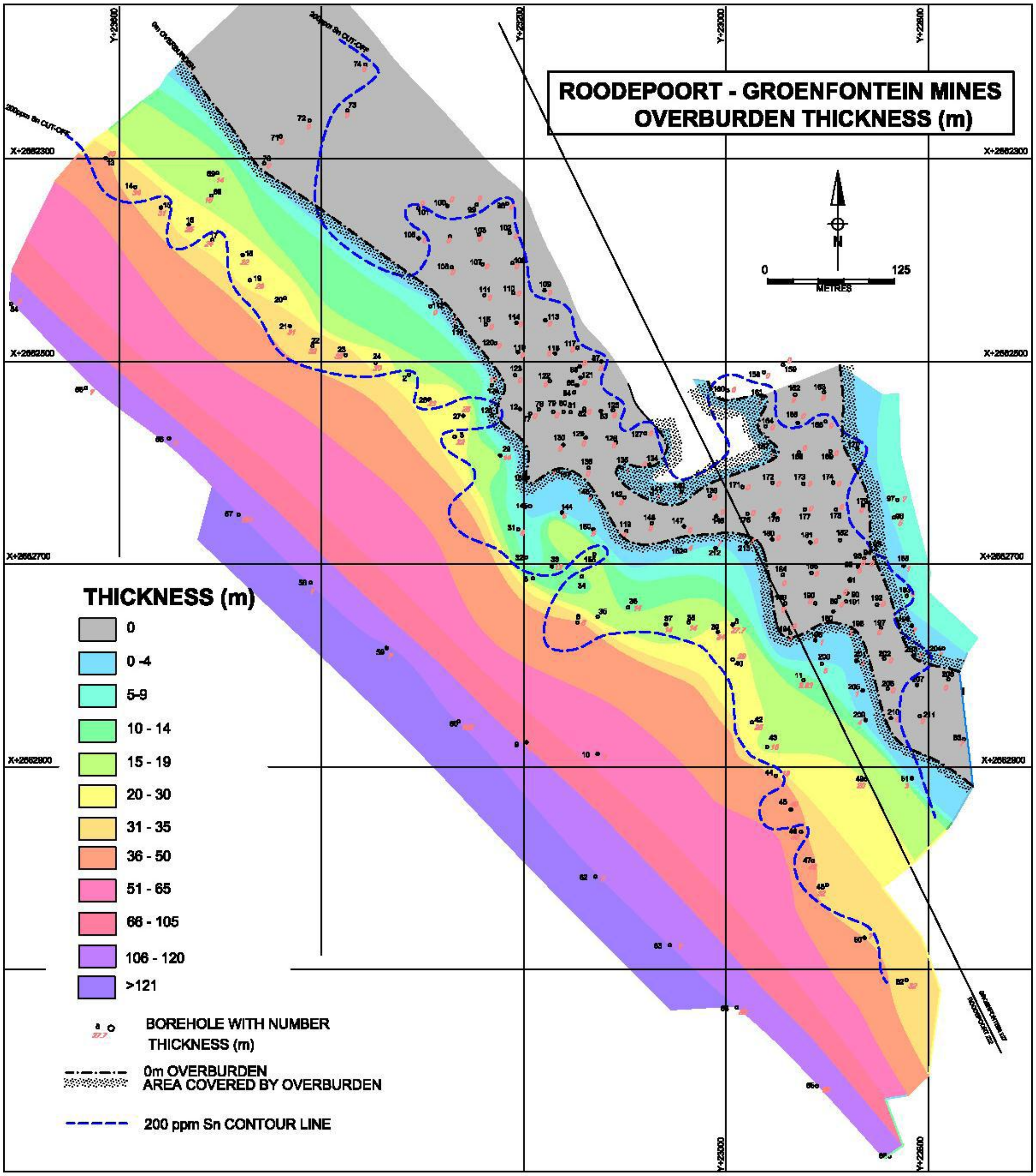

Fig 2.4: Overburden thickness (m) of the disseminated mineralisation zone

2.1.5 Composite map

A composite map showing tin content in relation to the axis of grade (grade isochon), tin content (isochon) and thickness (isopach) is shown in Fig 2.5. The axes show areas of thickest development of mineralisation. The tin content is also shown in relation to 200ppm cut-off. The extent of exposed mineralisation as well as the extent of area with underground workings is shown in Figure 2.5. As noted this reveals a close relationship between thickness and grade.

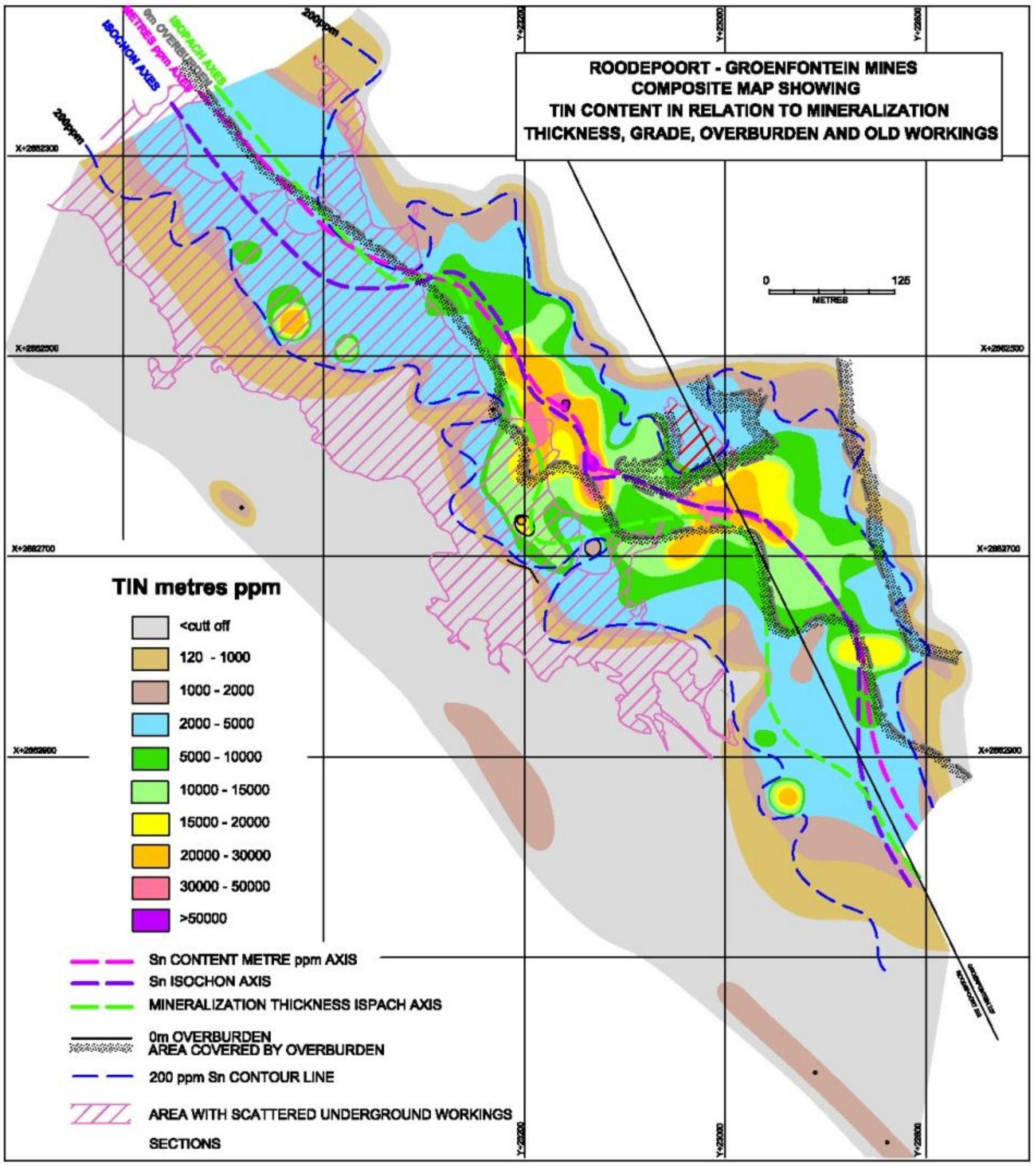

Fig 2.5: Composite map showing tin content and axis of highest tin content in relation to the axis of thickest mineralisation and highest tin grade. A close correlation is evident

2.2 Cross and longitudinal section portrayals of mineralisation

The position of each cross and longitudinal section is indicated on a locality map in Fig 2.6. The geology and mineralised zones are portrayed on the six cross-sections constructed from NE to SW and labelled AA^{I} to FF^{I} (Figs 2.7 to 2.12). Three NW trending longitudinal sections (GG^{I} to II^{I}) constructed from NW to SE are also portrayed (Figs 2.13 to 2.15). The cross-sections further show that the mineralisation forms a broad, dome-like manto, the crest of which has been removed by erosion. The general dip of the main mineralisation, SW limb of the manto is towards south west.

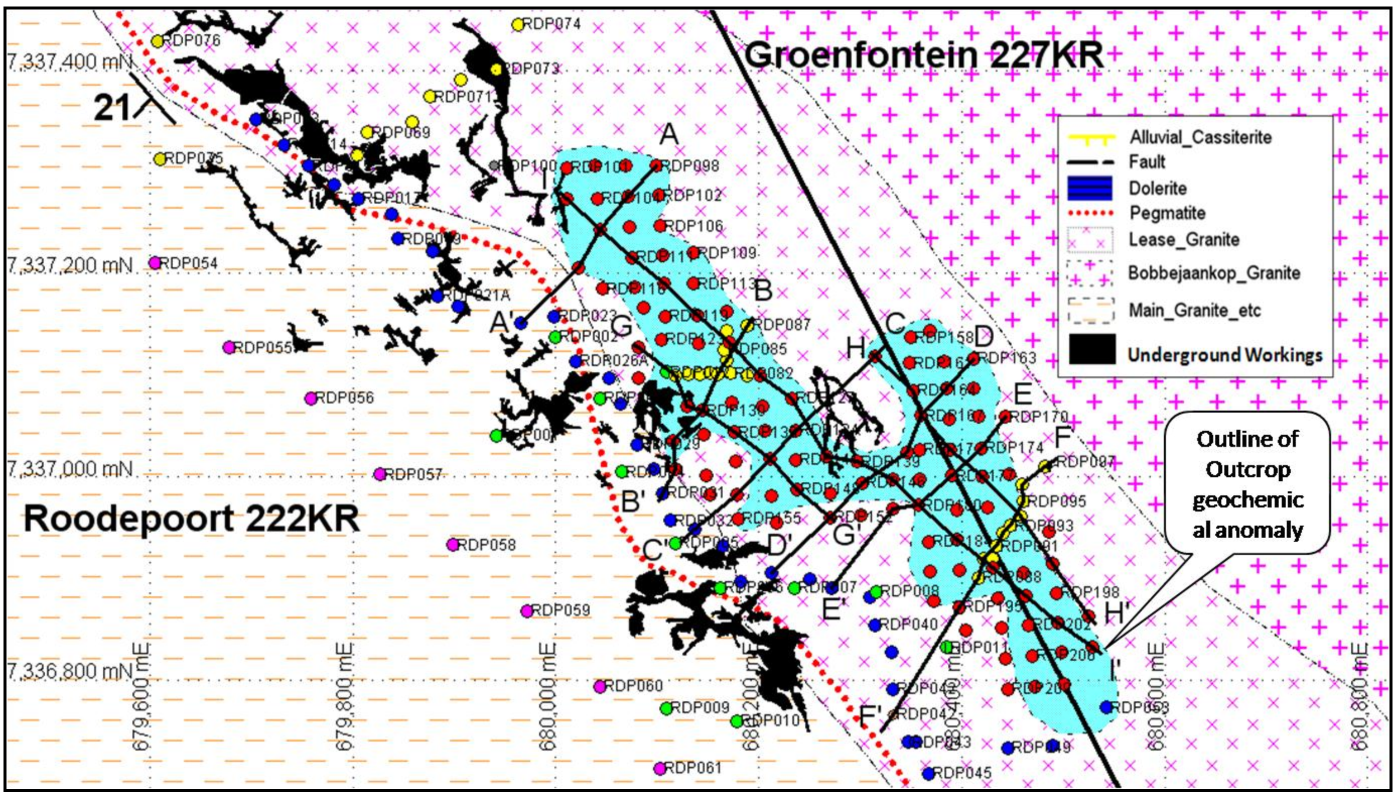

Fig 2.6: Plan view of cross sections (AA^{I} – FF^{I}) and longitudinal section (to GG^{I} -II^{I})

2.2.1 Cross section A-A^{I}

This is the furthest NW of the section lines and comprises at least a 140m wide zone of exposed and mineralised (>120) Lease Granite. The mineralisation continues down dip to the SW and is directly overlain by barren Main Granite which in turn is overlain by a Pegmatite. The Pegmatite is overlain by the Main Granite (Fig 2.7). Drill holes RDP103 and RDP108 did not fully penetrate the mineralisation. Infill drilling will be required to improve certainty in the thickness of the mineralisation.

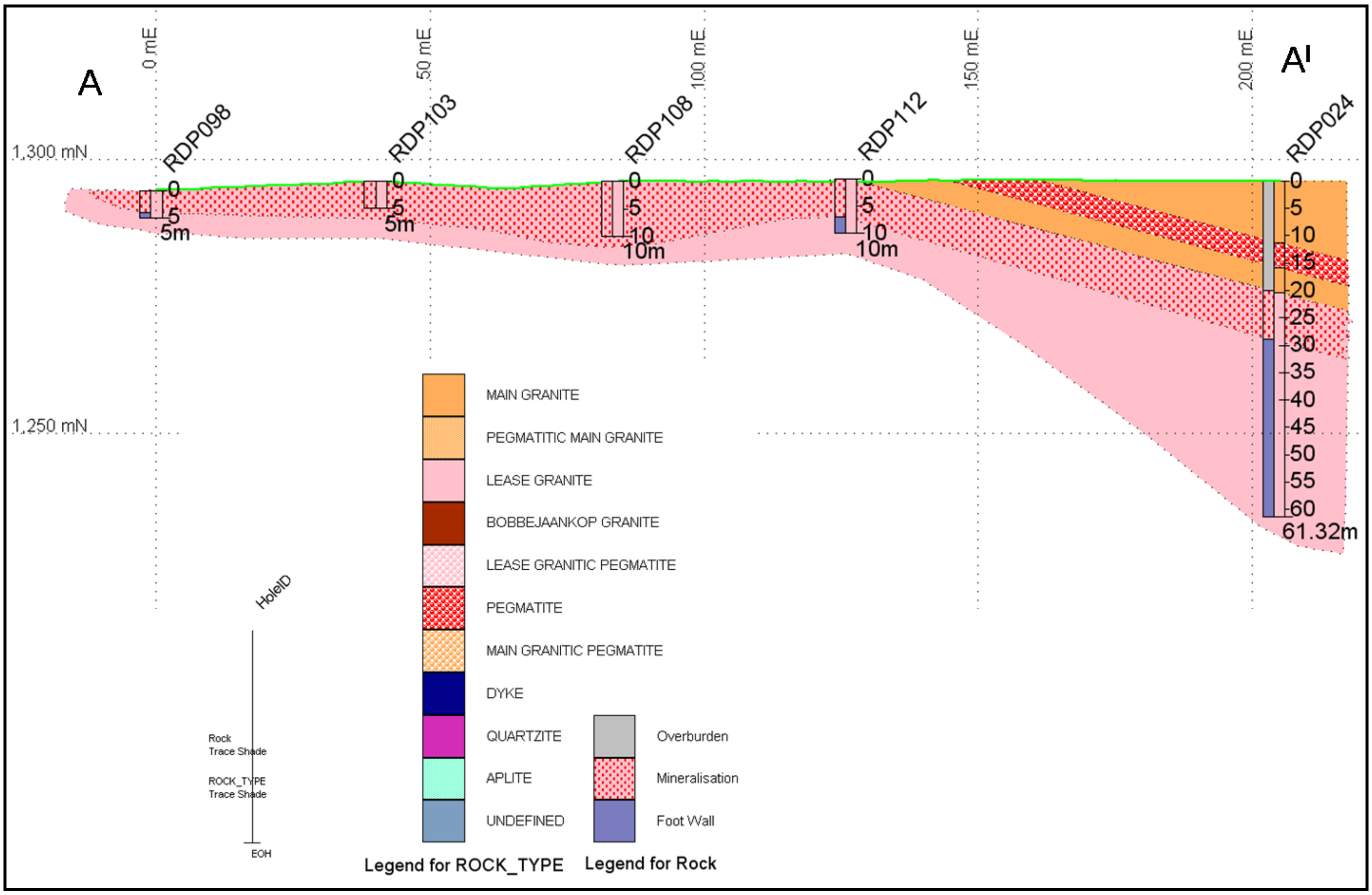

Fig 2.7: Section AA^{1}

2.2.2 Cross section B-B^{I}

This is the second furthest NW of the section lines. It comprises of at least 160m wide zone of mineralised (>120 ppm Sn) Lease Granite. The mineralisation continues down-dip and is directly underlain and overlain by barren Lease Granite in RDP145 and by Main Granite in RDP031 (Fig 2.8). Drill holes RDP087, RDP084 and RDP 081 did not fully penetrate the full mineralisation and further infill drilling will be required to ascertain the full mineralisation thickness in this zone.

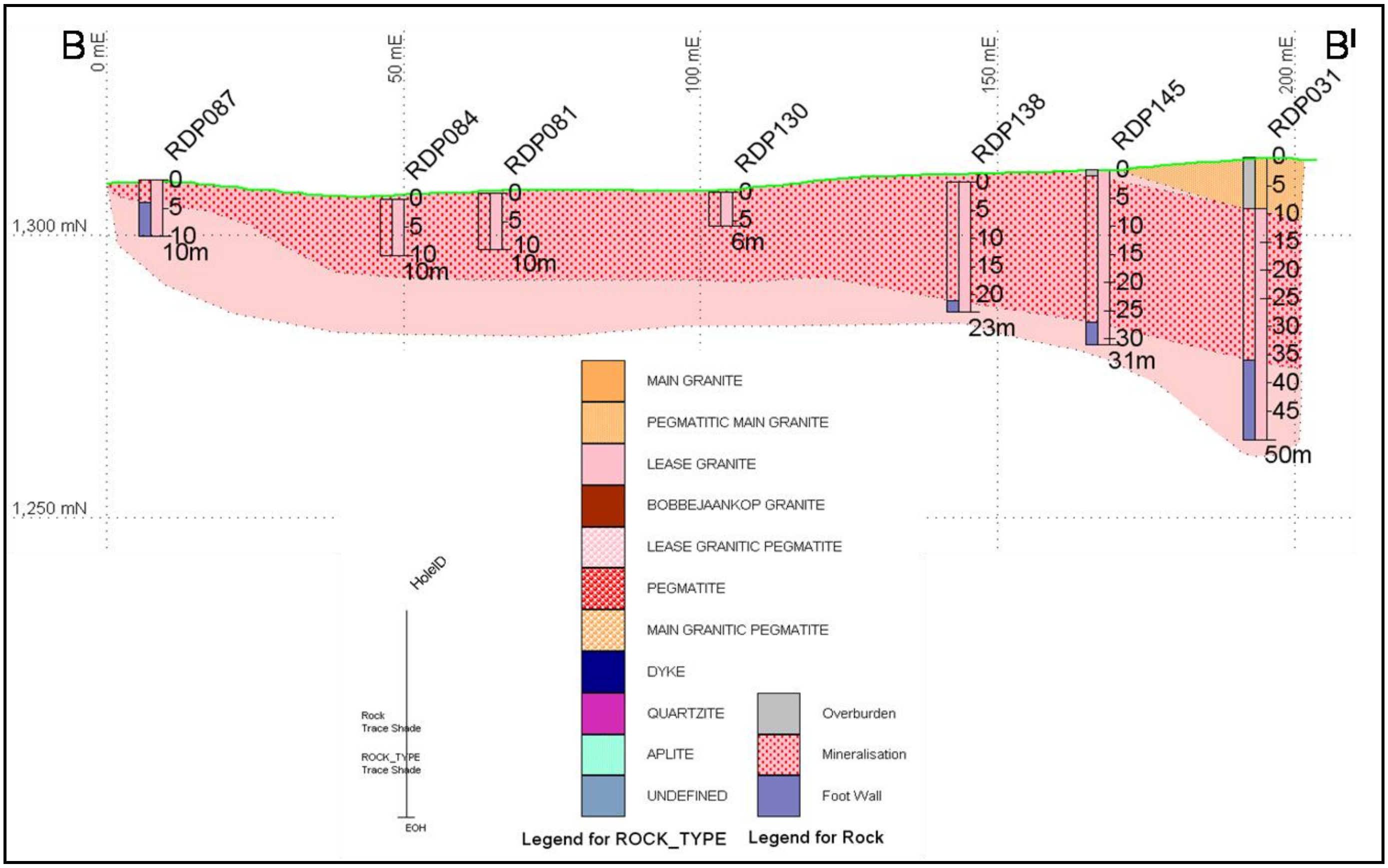

Fig 2.8: Section BB^{I}

2.2.3 Cross section C-C^{I}

This is the third furthest NW of the section lines which comprises of at least 180m wide zone of exposed mineralised (>120 ppm Sn) Lease Granite. The mineralisation continues down-dip to beyond 12m below surface and is directly overlain by barren Lease Granite which in turn is overlain by a Main Granitic Pegmatite zone (Fig 2.9). Drill holes RDP 134, RDP142 and RDP150 did not fully penetrate the mineralisation. Infill drilling will be required to improve knowledge about the thickness of the mineralised Lease Granite.

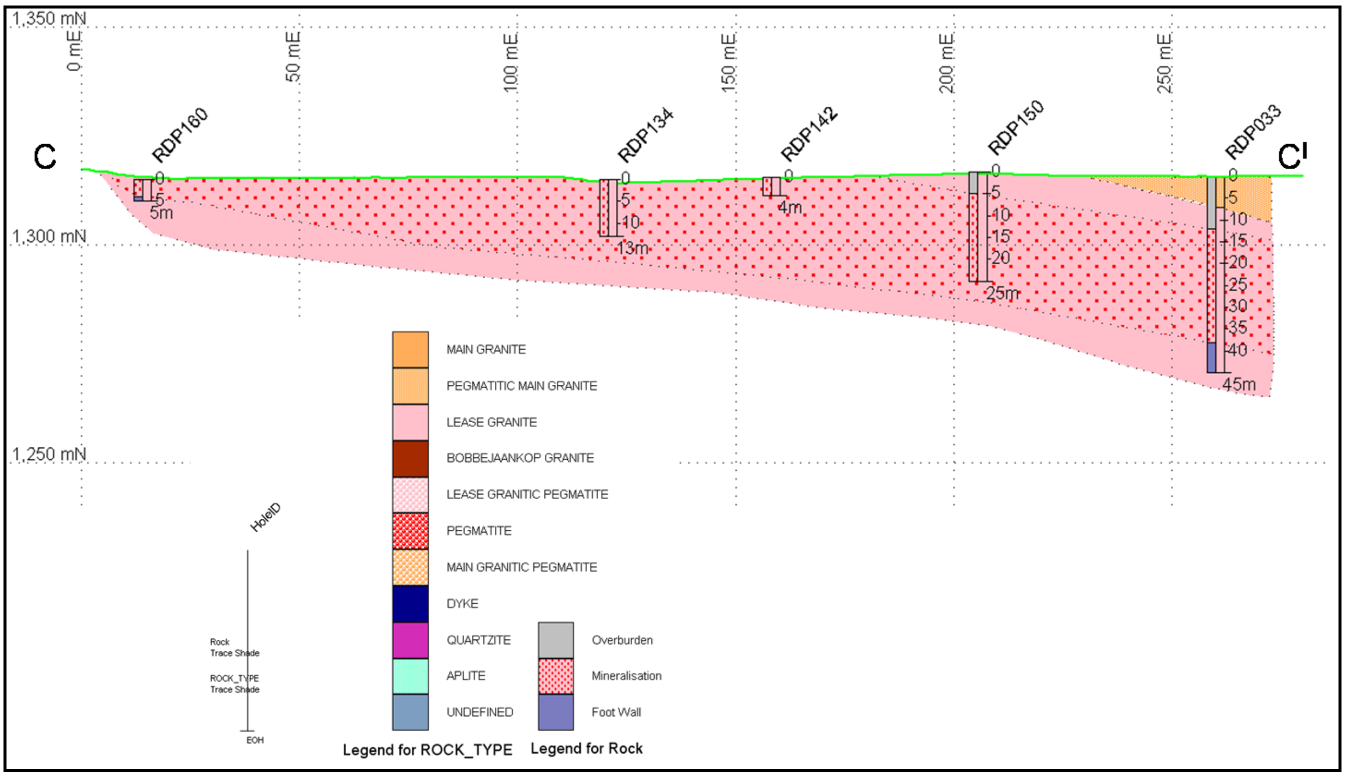

Fig: 2.9 Section CC^{I}

2.2.4 Cross section D-D[I]

This section line traverses the south-eastern area which comprises a 210m wide zone of mineralised (>120ppm Sn) Lease Granite. The mineralised zone is directly overlain by barren Lease Granite in RDP152 and by a Lease Granitic Pegmatite (Fig 2.10). The mineralised zone continues down-dip to below 15m below surface. The exposed mineralised Lease Granite extends for about 200m in this section but there is only one borehole, RDP 165 that fully penetrated the mineralisation. Further infill drilling will be required to ascertain the full thickness of the mineralised Lease Granite.

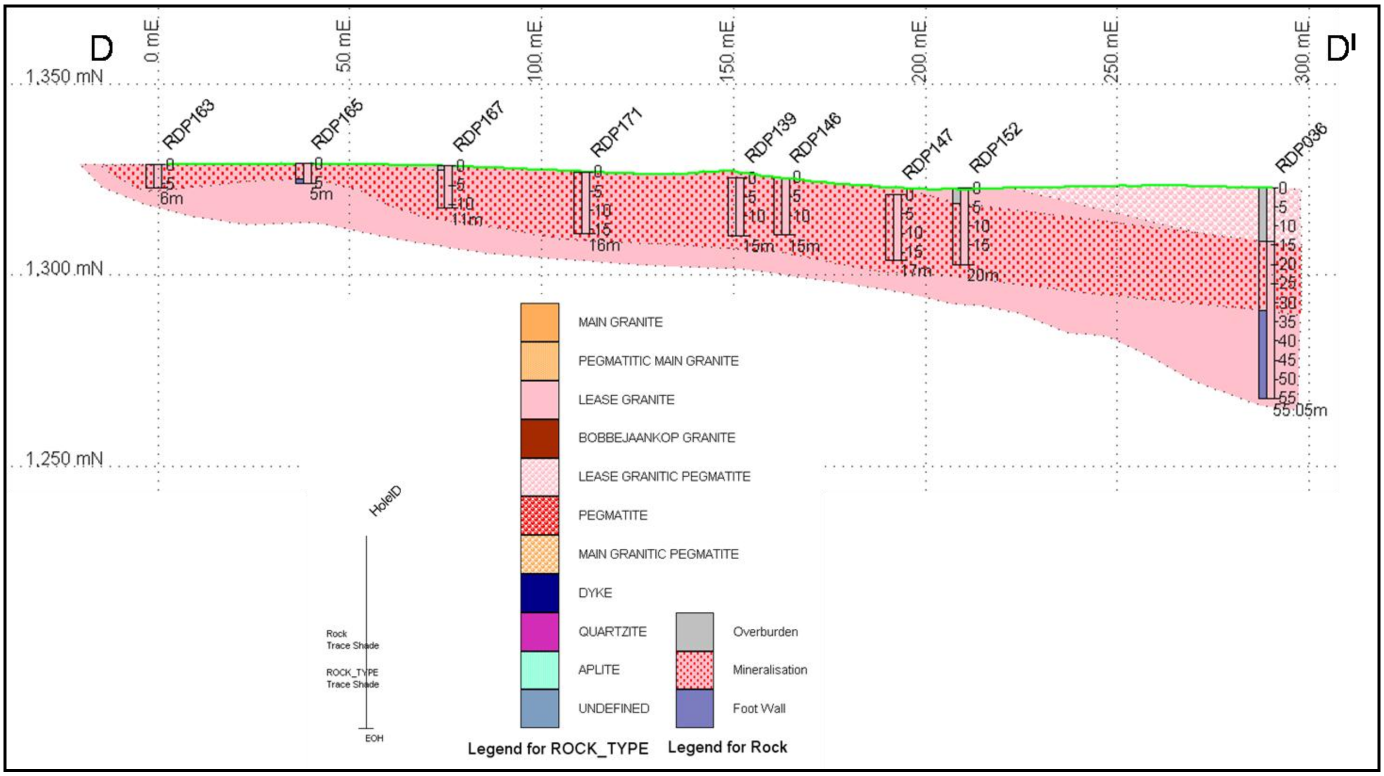

Fig 2.10: Section DD[I]

2.2.5 Cross section E-E^{I}

This section line traverses the southern portion of the surface mineralisation and comprises of a 104m wide zone of mineralised (>120ppm Sn) Lease Granite. It is directly overlain by barren Lease Granite which in turn is overlain by a Lease Granitic Pegmatite (Fig 2.11). The mineralisation continues down dip to below 20m below surface and dips shallowly to the NW in RDP170. Only RDP177 fully penetrated the mineralisation. Infill drilling will be necessary if the full thickness of the mineralised Lease Granite is to be known with higher level of accuracy.

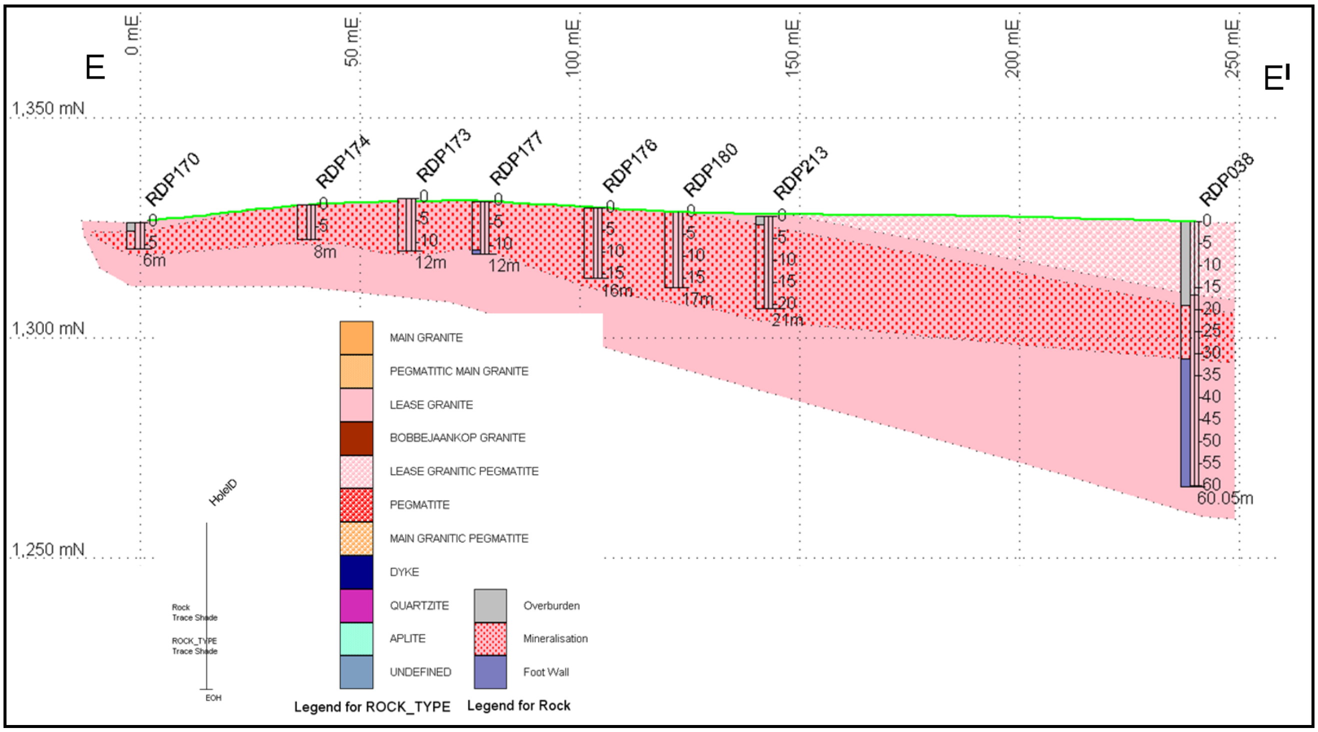

Fig 2.11: Section EE^{I}

2.2.6 Cross section F-F^{I}

This is the furthest SE of the section lines. It comprises of 115m wide zone of mineralised (>120 ppm Sn) Lease Granite. The mineralisation continues down-dip and is directly overlain by barren Lease Granite which in turn is overlain by the Main Granite (Fig 2.12). The mineralisation tapers off towards NE in RDP095 and RDP096 and the mineralisation was not intersected in RDP097. The boreholes on the exposed portion of the mineralised Lease Granite did not fully penetrate the mineralisation. Thus infill drilling is recommended to improve knowledge about the thickness of the mineralised Lease Granite.

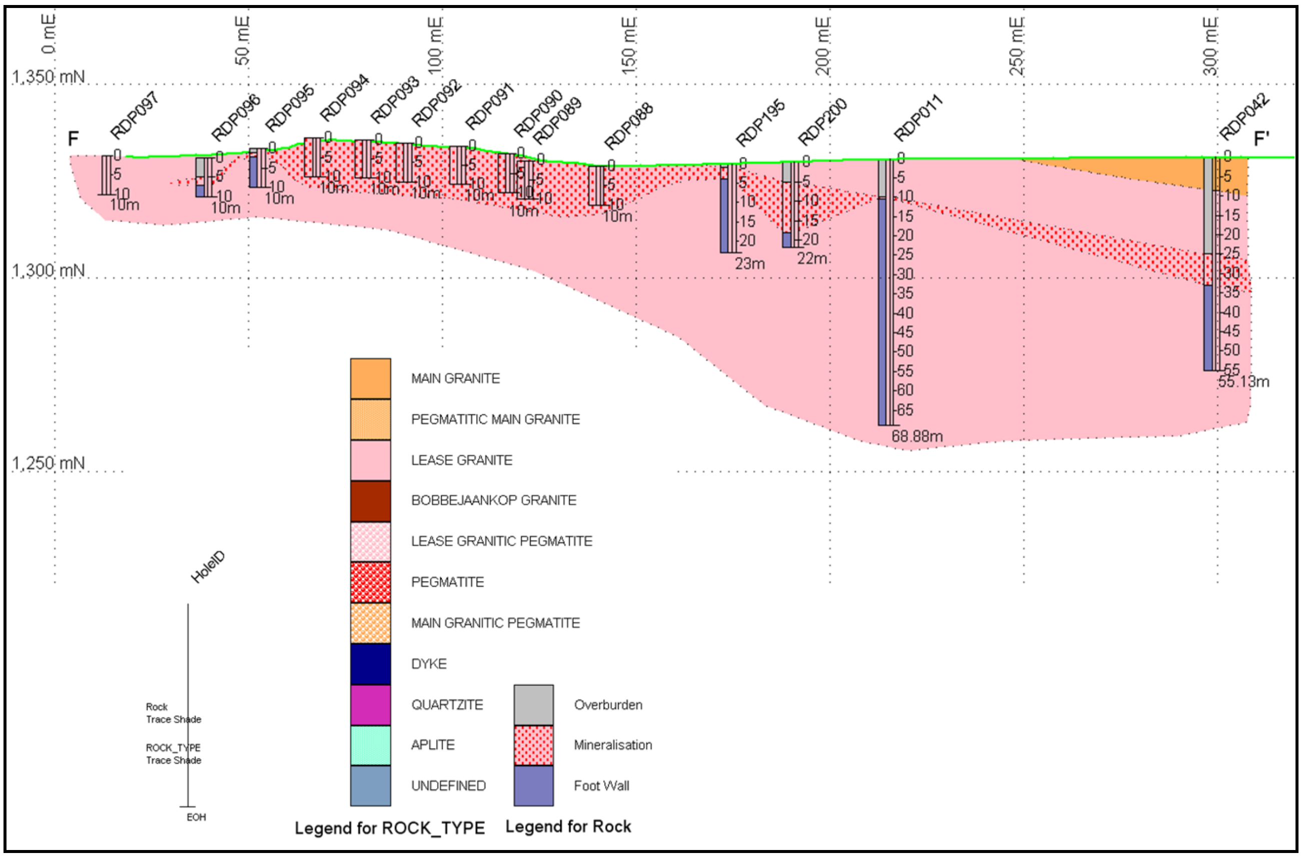

Fig 2.12: Section FF^{I}

2.2.7 Cross section G-G^{I}

This is the furthest SW longitudinal section and shows a 200m long mineralised sector in Lease Granite. The mineralisation dips towards NW to below 8m in RDP124 and also towards SE to below 5m in RDP 152 (Fig 2.13). The drill-holes did not fully penetrate the mineralised Lease Granite along the entire section. The mineralisation thickness will require verification through further drilling along the entire section to ascertain the full mineralisation thickness. Drill holes RDP130 and RDP142 were too shallow to add value as far as mineralisation thickness is concerned. Further infill drilling is recommended.

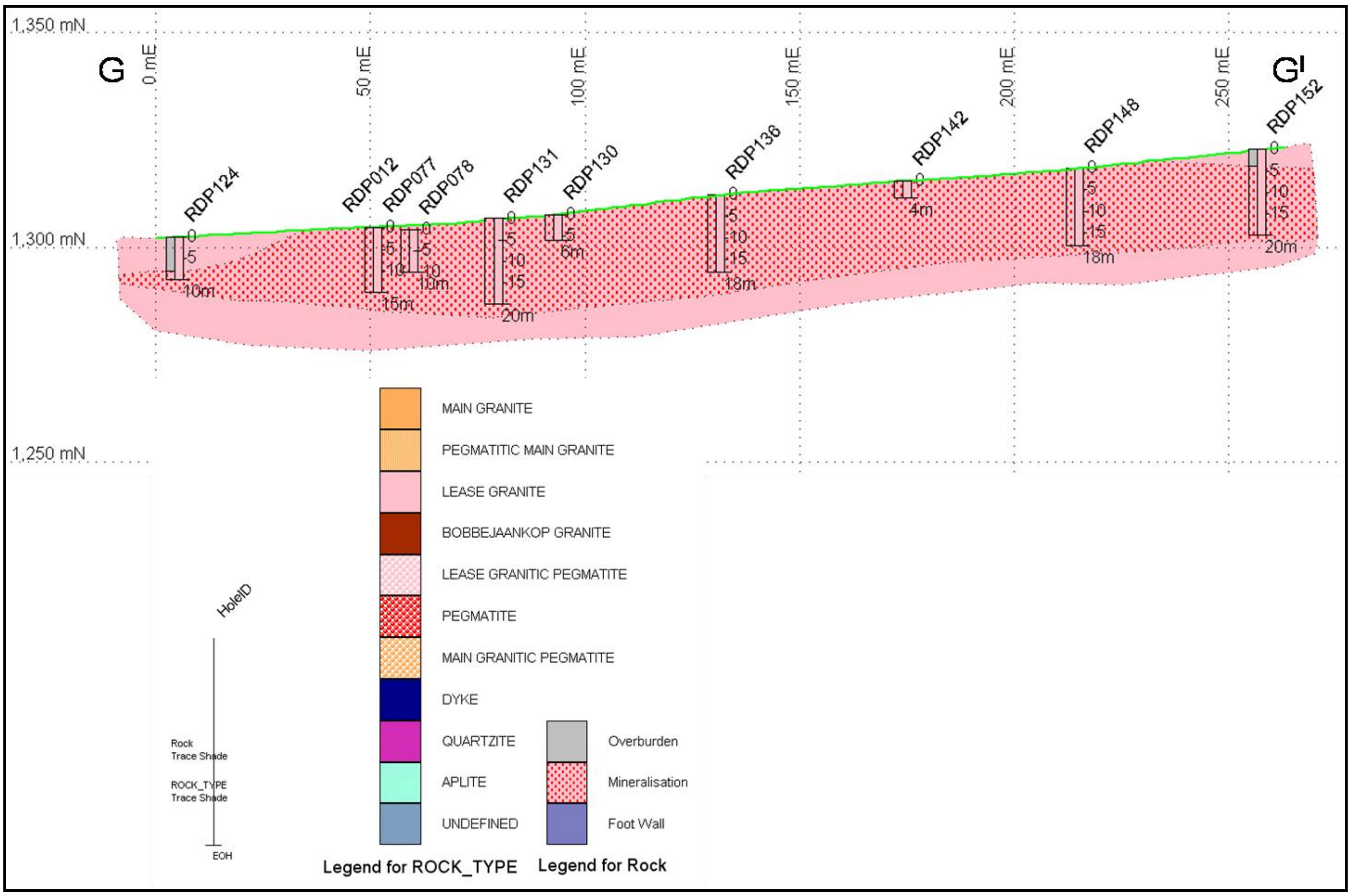

Fig 2.13: Section GGI

2.2.8 Cross section H-H^{I}

This is the furthest NE longitudinal section and is also towards the NE edge of the mineralised area. The exposed mineralisation extends for about 240m in Lease Granite. The mineralised zone is overlain by thin (<5m thick) barren Lease Granite from about 15m SE of RDP94, for at least 90m towards the SE through RDP193 and towards RDP204 (Fig 2.14). Only 3 boreholes intersected the full mineralisation, thus, additional drilling will be required to fully penetrate the mineralised Lease Granite.

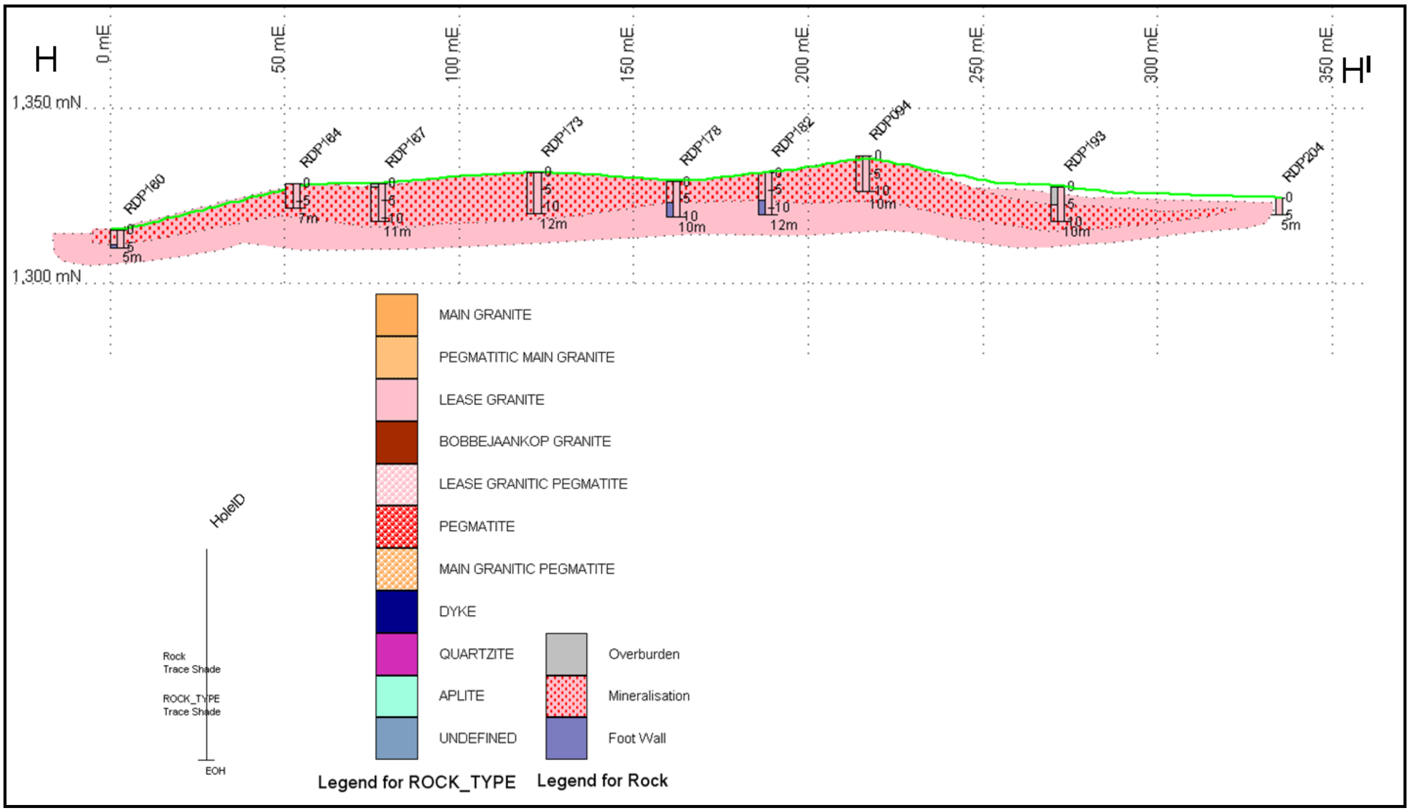

Fig 2.14: Section HH^{I}

2.2.9 Cross section I-I^I

This is the middle and longest 670m longitudinal section and mineralised Lease Granite is exposed along the whole section (Fig 2.15). Only 3 boreholes intersected the full mineralisation, thus, additional drilling will be required to fully penetrate the mineralised Lease Granite.

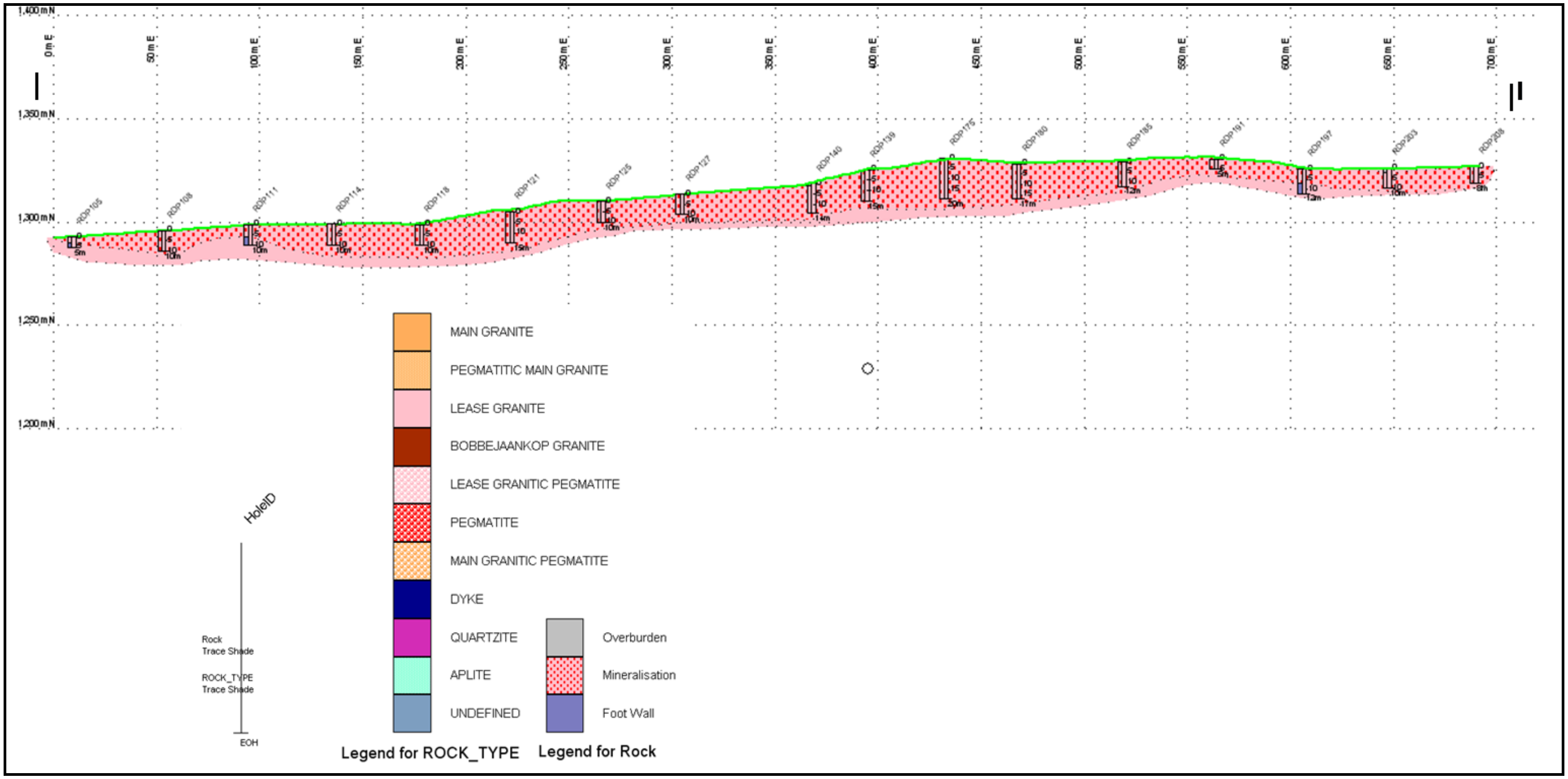

Fig 2.15: Section II^I

2.3. Conclusion

The modelling of the Roodepoort - Groenfontein tin mineralisation has revealed the shape and extent of potentially economic tin resource which is exposed on surface and is about a kilometre long, about 200m in width and a maximum thickness of 30m. The mineralisation dips to the SW at between 15-20^{o}. The models show that the mineralisation thins-off in all directions and is elongated in the NW-SE direction. The high tin grade and content is mainly in the central portion of the elongate domical structure with two distinct ‘eyes’, one in SE and the other in the NW central portion of the mineralised Lease Granite. Most drill-holes that were collared on the exposed mineralised Lease Granite did not fully penetrate the full extent of mineralisation and the modelling of the mineralisation thickness was based on extrapolation and interpolation of known intersections. Additional drilling is recommended in order to ascertain the full extent of the thickness of mineralised Lease Granite.

CHAPTER 3: POLYGONAL ESTIMATION

3.0 Introduction

The polygons were generated in MapInfo software and are such that the borehole or sample is polygon-centered and the boundary of each polygon is equidistant between neighbouring boreholes. The size of the polygons depends on the distance separation between neighbouring boreholes, the closer the boreholes are to each other the smaller the polygon and vice-versa. The mineralised area with underground workings was included in the resource estimation since the previous underground mining targeted only the isolated tin-rich pipes. It is estimated that a maximum of 5% by volume of mineralised granite was removed by underground operations. Thematic maps showing mineralisation thickness (m), tin grade (ppm), tin content (m*ppm) and overburden thickness are shown in Figs 3.1 to 3.4. The mineral resource under study constitutes polygons that have overburden of less than 30m and these are shown to the East of the 30m contour in Figs 3.1-3.4. The extent of underground workings is also shown in the figures. The polygons that were bisected by the 30m contour line and the portions of polygons with overburden of less than 30m are shown in Fig 3.5. The polygons used for estimation are listed in Appendix 2.

3.1 Mineralisation thickness (m)

As noted and described in Chapter 2, the disseminated mineralisation is thickest (15-30m) in the central portion of the project area where a zone extending for about 440m along strike by 50m in width is present (Fig 3.1). This forms the core of a more extensive zone of mineralisation, which, using a 5m thickness cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 3.1) where the mineralised zone has not been fully investigated. These areas could offer an additional tin resource (Fig 3.1).

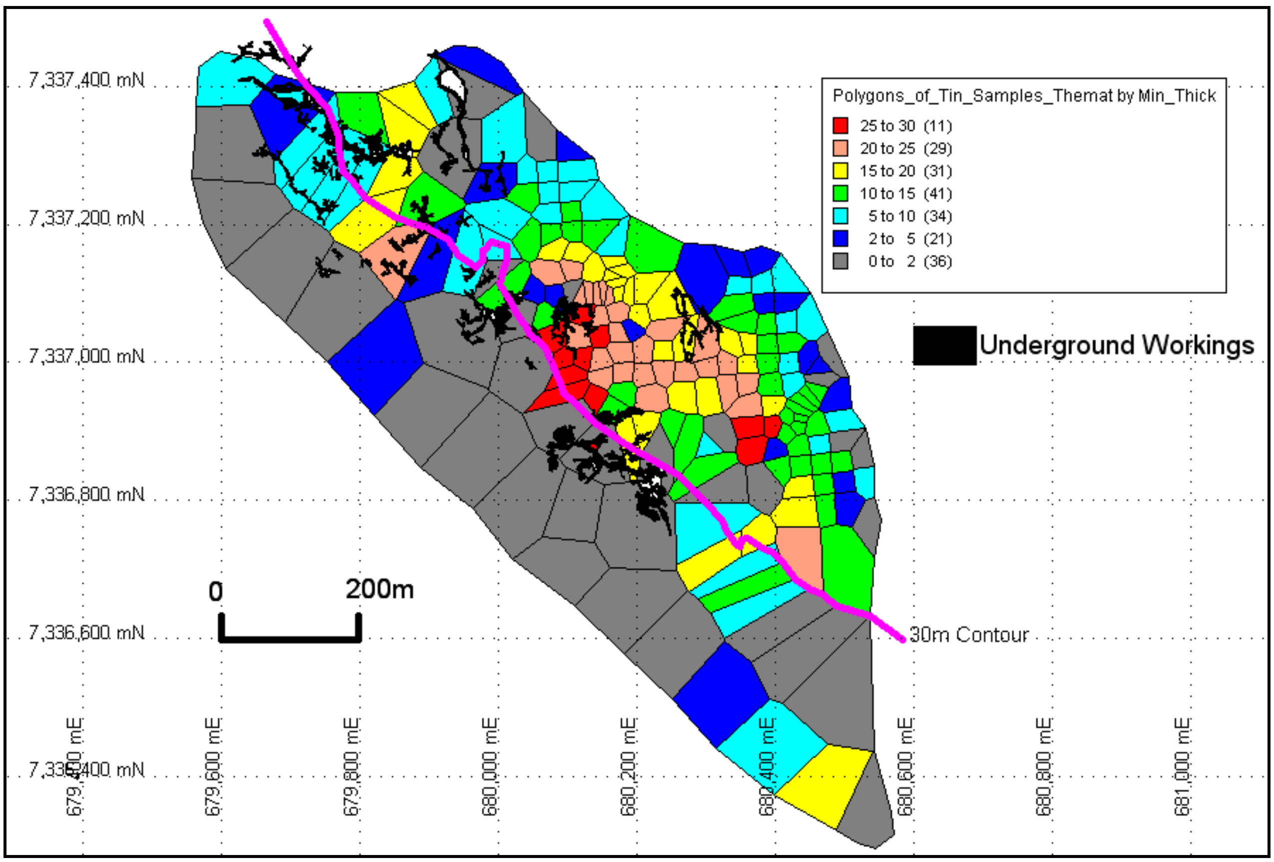

Fig.3.1: Polygons showing mineralisation thickness (m)

3.2 Grade (ppm Sn)

The mineralisation grade is highest (>1,000 ppm Sn) in two distinct zones, one in the central south-eastern and the other in the central north-western portion of the mineralisation manto and these are shown in red circles in Fig 3.2. The grade of greater than 500ppm Sn encompasses the higher grade areas, and extends for about 600m along strike by 90m in width, punctuated by patchy portions of low grade, 200-500ppm Sn (Fig 3.2). In line with the observations in Chapter 2, the area forms the core of a more extensive zone of mineralisation, which, using a 200ppm cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 3.2) where the mineralised zone has not been fully investigated and these areas could offer an additional tin resource (Fig 3.2). The 30m contour line is shown and demarcates the area used for resource estimation to the east.

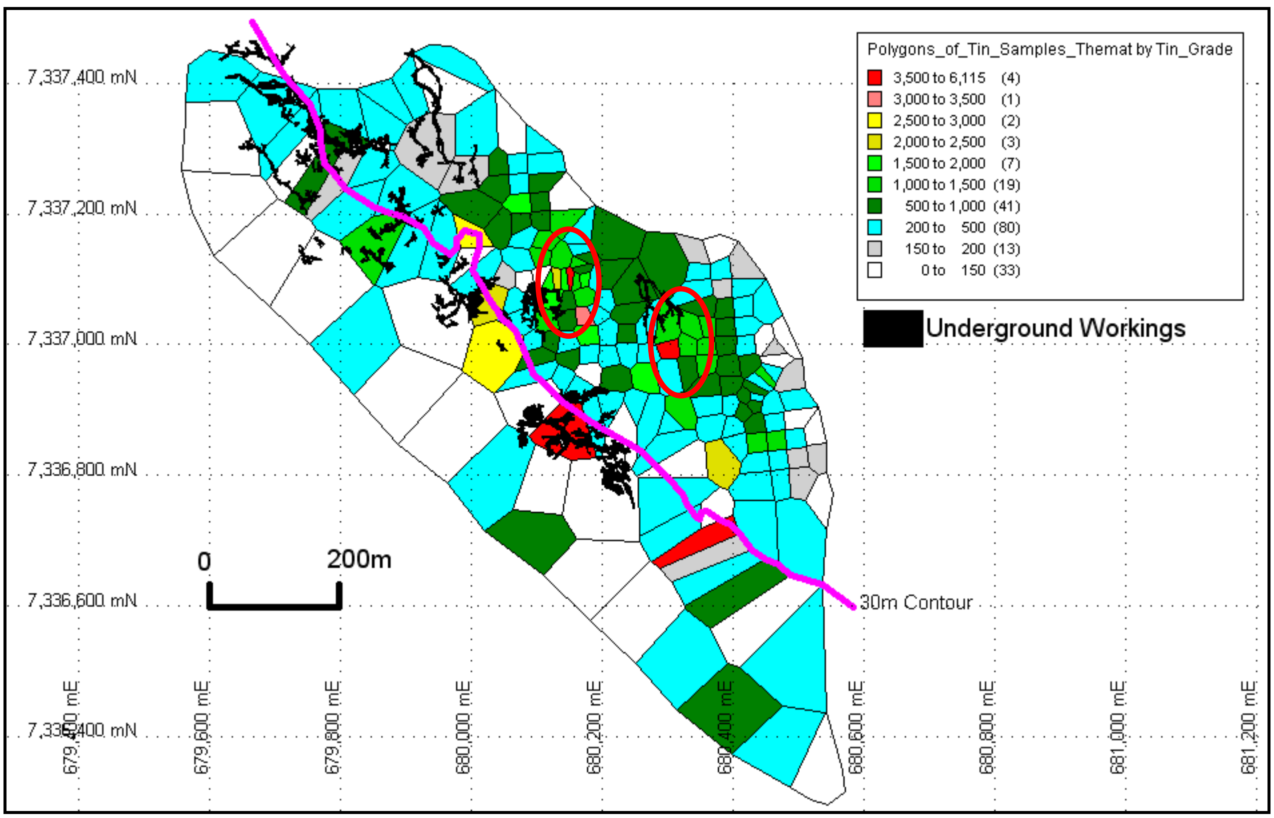

Fig.3.2: Polygons showing mineralization grade (ppm Sn)

3.3 Tin Content (m*ppm Sn)

As with the observed grade distribution, two distinct zones of high tin content greater than 15,000m*ppm can be observed, one in the south-eastern and the other in the north-western portion of the investigation area (Fig 3.3). The content greater than 5,000m*ppm Sn extends for about 600m along strike by 90m in width (Fig 3.3). This forms the core of a more extensive zone of mineralisation, which, using a 2,000m*ppm cut-off extends for over 1,000 metres along the strike and averages about 200m in width. It is open ended along strike to both the NW and SE (Fig 3.3) where the mineralised zone has not been fully investigated and again the areas could offer an additional tin resource (Fig 3.3).

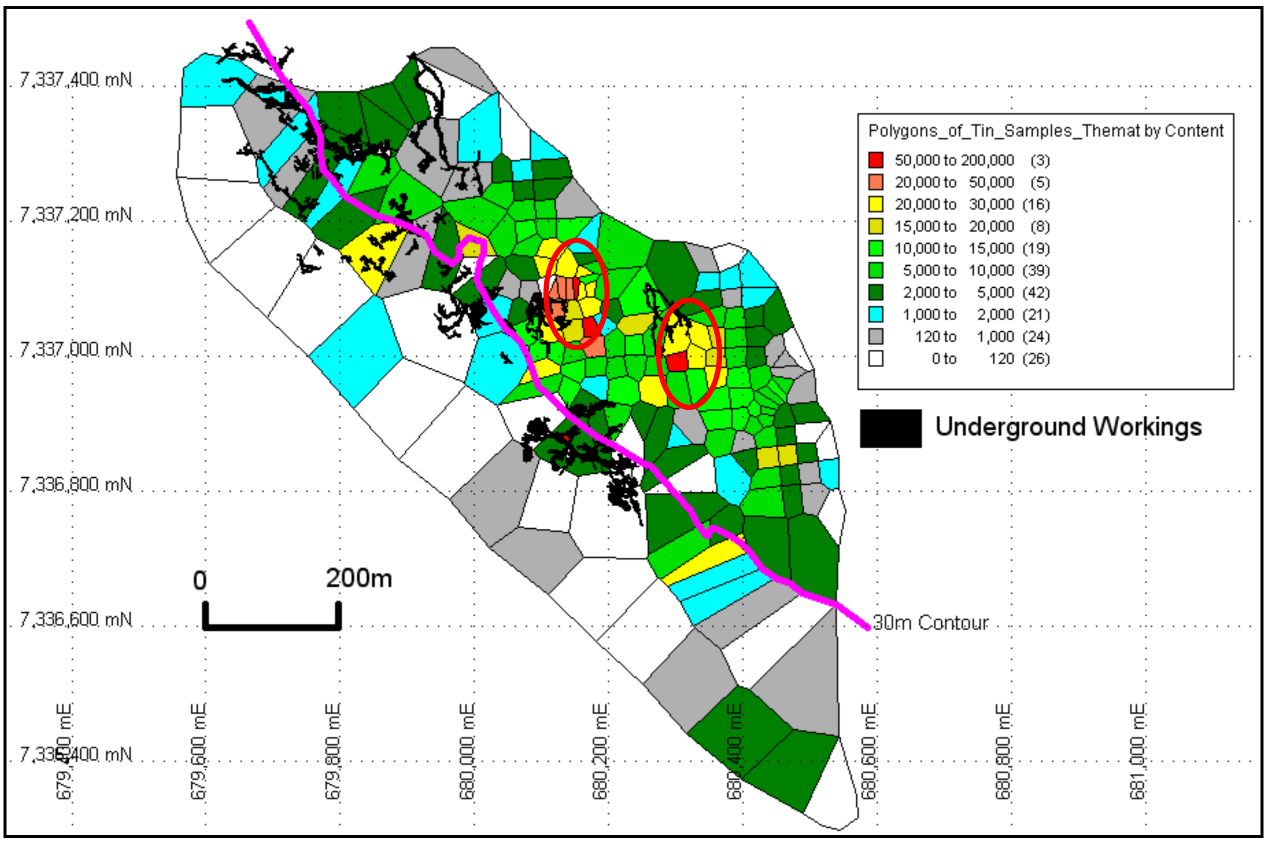

Fig 3.3: Polygons showing tin content (m*ppm Sn)

3.4. Overburden thickness (m)

The mineralised tin granite is largely exposed on the surface, white polygons to the east of the 30m overburden line (Fig 3.4) with few scattered covered areas with less than 5m overburden, which could be attributed to superficial cover rather than hard rock.

The overburden thickness increases down dip. The 30m overburden thickness is shown and all polygons to the west of the line are under thicker overburden cover of more than 30m (Figure 3.4). The white polygons to the west of the 30m contour line are not mineralised and therefore no overburden information could be plotted. An overburden of 3.25 million tons has to be removed to expose the mineralised Lease Granite.

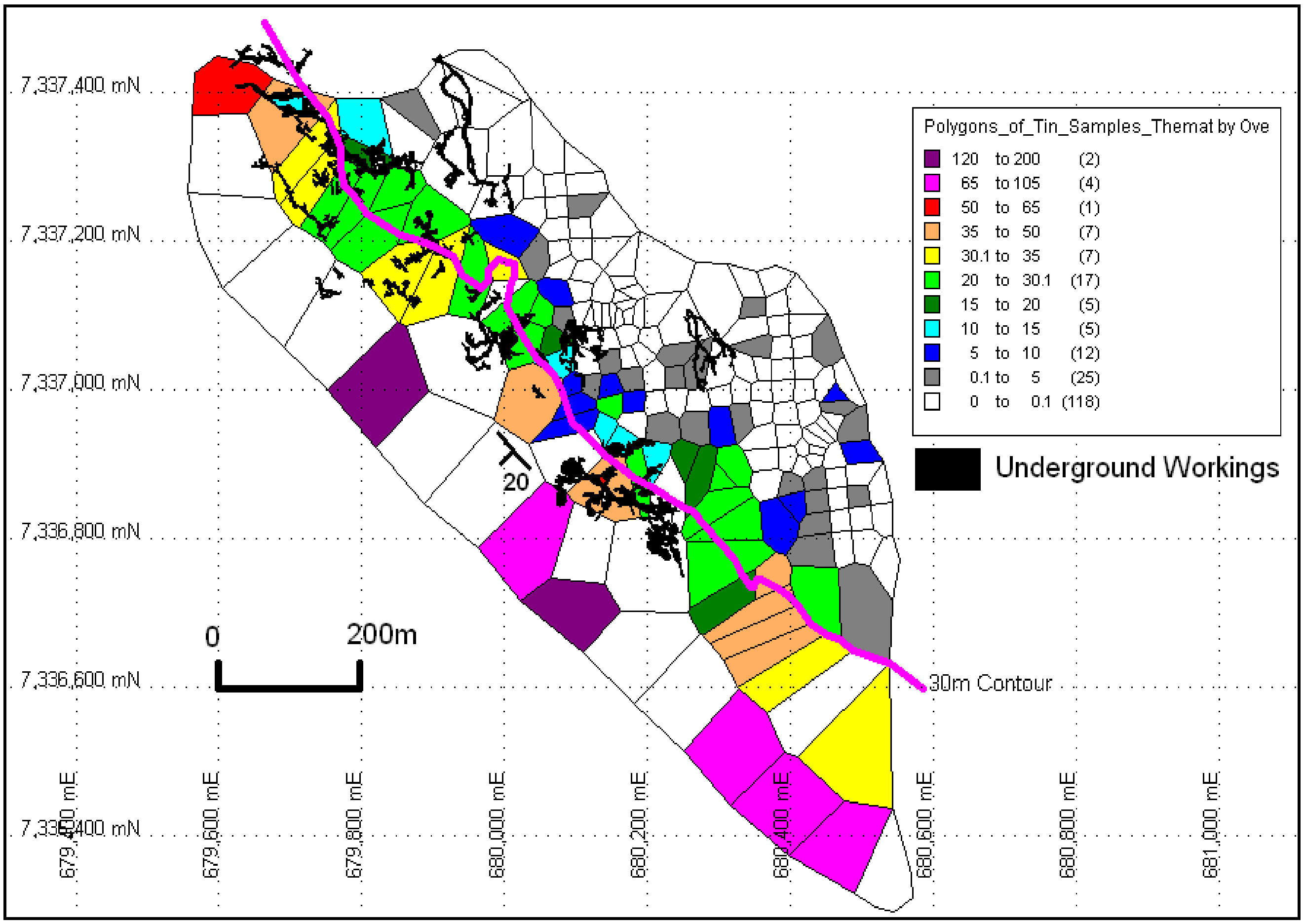

Fig 3.4: Polygons showing overburden thickness (m)

3.4.1 Overburden tonnage vs resource tonnage

Table 3.1: Waste and Mineralisation tonnage for area covered by overburden

	Waste Tonnage	Resource Tonnage
Total Volume (m^3)	1,266,496	1,351,237
Density of Lease Granite(Kg/m^3)	2,570	2,570
Waste (kg)	3,254,895,852	3,472,678,027
Waste (Ton)	**3,254,896**	**3,472,678**

Density was assumed to be more or less the same for both waste and mineralised resource since the mineralisation is of low grade disseminated nature. Considering an overburden limit of 30m and a cut-off grade of 200ppm the overburden waste to resource ratio is 0.9 (Table 3.1).

3.5 Cut-off grades and tonnages

The 200ppm Sn cut-off and 500ppm Sn cut-off have been considered for the outcropping portions of the ore body as well as for parts of the ore body with an overburden thickness less than or equal to 30m. The mineral resource figures were calculated using the polygonal method. The polygons used for a 200ppm Sn cut-off and those used for 500ppm Sn cut-off are shown on the thematic map in Fig 3.5.

3.5.1 Grade: 200ppm Sn cut-off

The number of polygons above 200ppm Sn cut-off grade is 147. The polygons are inter-connected and this demonstrates that the 200ppm cut-off grade gives mineralisation continuity (Fig 3.5). For a cut-off grade of 200ppm Sn and for mineralisation with overburden of less than 30m, the average grade is 813.05 ppm Sn with a tonnage of 6,255,213.49 tons Sn when 5% is discounted for the area with underground workings. The tin tonnage would be 5,085.81 tons Sn.

3.5.2 Grade: 500ppm Sn cut-off

The number of polygons above 500ppm Sn cut-off grade is 74. The polygons for 500ppm Sn cut-off are shown in Fig 3.5 and show clustered blocks which, if adhered to rigorously, would require a selective mining approach. For a cut-off grade of 500ppm Sn and ore with overburden of less than 30m, the average grade is 1,288.28 ppm Sn with ore tonnage of 2,610,307.14 tons Sn if 5% is discounted for the area with underground workings. The tin tonnage would be 3,362.82 tons Sn.

Table 3.2: Polygonal resource estimates for 200 ppm Sn and 500 ppm Sn cut-off grades

Polygonal Estimate	200ppm Sn Cut-off		500ppm Sn Cut-off	
	Resource	5% Discounted	Resource	5% Discounted
Total Volume	2,562,037	2,433,935	1,069,1401	1,015,684
Density of Lease Granite (Kg/m^3)	2,570.00	2,570.00	2,570.00	2,570.00
Ore kg	6,584,435,251	6,255,213,488	2,747,691,731	2,610,307,144
Ore Ton	6,584,435	6,255,213.49	2,747,691.73	2,610,307.14
Grade	813.05	813.05	1,288.28	1,288.28
Tin grammes	5,353,489,414	5,085,814,943	3,539,806,700	3,362,816,365
Tin Kg	5,353,489	5,085,815	3,539,807	3,362,816
Tin Ton	5,353.49	5,085.81	3,539.81	3,362.82

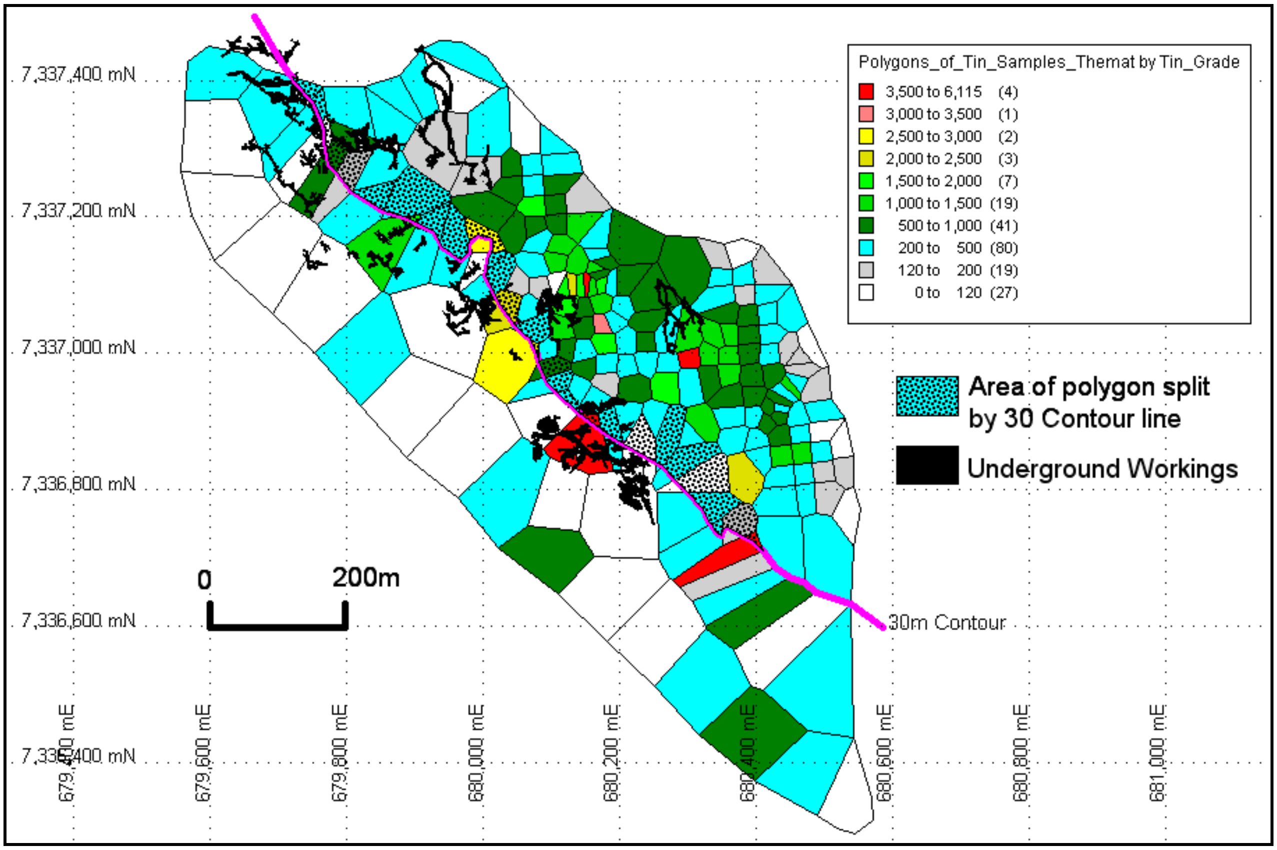

Fig 3.5: Polygons used for resource estimation, area with underground workings is also shown

Table 3.3: Summary statistics for the 147 polygons above 200 ppm Sn cut-off

	Tin Grade(ppm)	Overburden Thickness(m)	Mineralization Thickness(m)	Area(m2)	Volume	Content(m*ppm)
Count	147	147	147	147	147	147
Sum	119,518.67	912.23	2,091.67	197,473	2,562,037	1,713,453.75
Max	6,114	49	30	7,915	101,525	122,280
Min	200	0	0.61	134	85	448
Average	**813.05**	6.21	14.23	1,343	17,429	11,656.15
Kurtosis	15.97	2.60	-0.93	9.47	8.14	23.63
Skewness	3.66	1.85	-0.04	2.72	2.31	4.29
Var	889,791.66	118.73	50.73	1,321,872.77	225,747,635.24	269,512,223.75
StDEv	943.29	10.90	7.12	1,149.73	15,024.90	16416.83
Cov	1.16	1.76	0.50	0.86	0.86	1.41

Table 3.4: Summary statistics for the 74 Polygons above 500 ppm Sn cut-off

	Tin Grade(ppm)	Overburden Thickness(m)	Mineralization Thickness(m)	Area(m2)	Volume	Content(m*ppm)
Count	74	74	74	74	74	74
Sum	95,333	356.85	1,119.05	78,469.52	1,069,140.75	1,383,281
Max	6,114	49	27	5,654.14	52,955.17	122,280
Min	503	0	0.61	134.15	84.92	793
Average	**1,288.28**	4.82	15.12	1,060.40	14,447.85	18,692.99
Kurtosis	8.90	5.50	-0.66	12.86	2.04	13.26
Skewness	2.88	2.52	-0.47	2.97	1.18	3.30
Var	1,311,180.43	122.81	48.92	691,116.33	103,754,019.08	429,284,424.59
StDEv	1145.07	11.08	6.99	831.33	10185.97	20,719.18
Cov	0.89	2.30	0.46	0.78	0.71	1.11

3.5.3 Grade-Tonnage curve for the tin resource

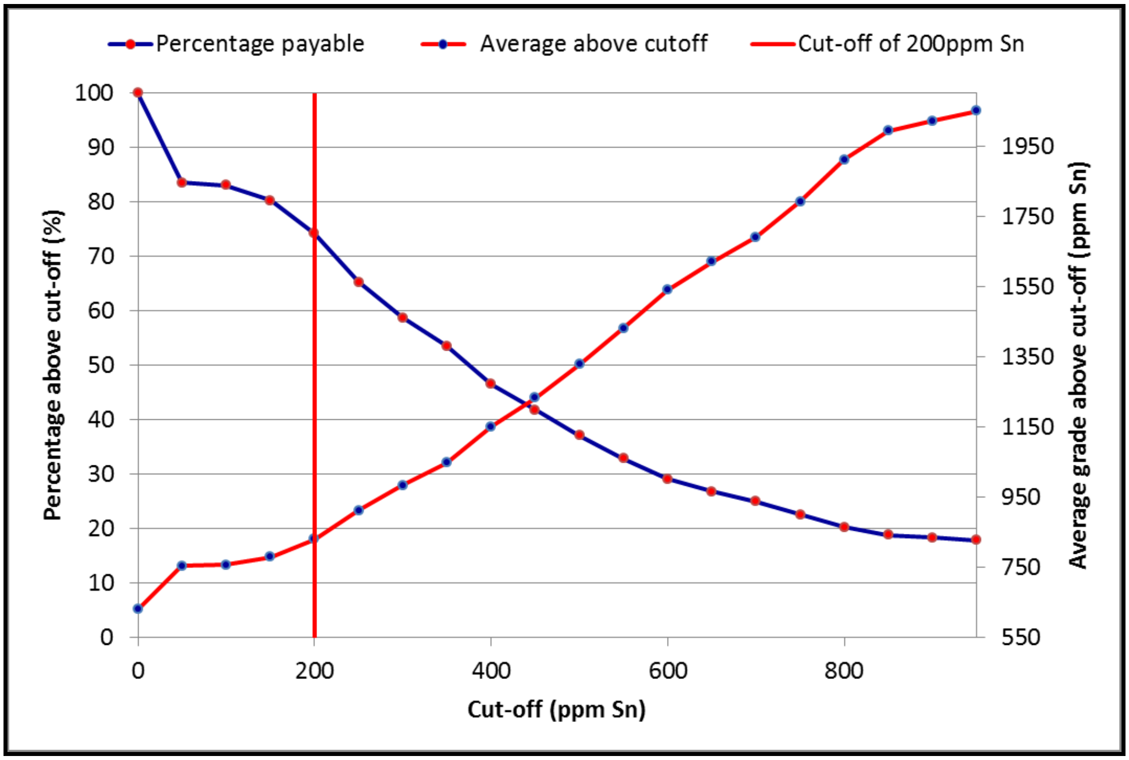

Fig 3.6: Grade-Tonnage curve for the tin mineralisation on Roodepoort-Groenfontein resource

The grade-tonnage curve in Figure 3.6 indicates that at 200ppm Sn cut-off, 75% of the resource tonnage would be available and the average grade would be 828 ppm Sn. The total tonnage for the tin resource is 8.7 million tones. Therefore, 6.5 million tons of tin resource is above 200ppm Sn. These numbers are in line with calculations in section 3.5.1.

Fig 3.6 shows that a cut-off grade of 500 ppm Sn would significantly reduce the resource tonnage, only 38% of the tonnage would be available for exploitation and average grade above cut-off would be 1327ppm Sn. These values are quite comparable with the analysis in section 3.5.2

3.6 Recommendations

The cut-off grades are summarised in Table 3.5 below.

Table 3.5: Scenario analysis for 200ppm Sn and 500ppm Sn cut-off.

Cut-off grade (ppm Sn)	Polygons used	Av grade above cut-off (ppm Sn)	Resource Tonnage (tons)	Tin Tonnage (tons)	Difference (Resource Tons)	Difference (Tin Tons)
200	147	813	6,255,213	5,086		
500	74	1,288	2,610,307	3,363	3,644,906	1,723

Increasing cut-off grade from 200ppm to 500ppm has the effect of reducing resource tonnage significantly by 3,644,906 tons ore and tin tonnage by 1,723 tons Sn. In this case, the cost benefit ratio will need to be calculated.

If mining at a 200ppm cut-off grade is considered, it should be justified that the income generated from an additional 1,723 ton Sn should offset costs incurred in mining and processing of an additional 3,644,906 tons ore. A feasibility study of the two options would be required. The cut-off grade of 200ppm offers attractive ore body continuity, which in turn would make mining much easier and possibly cheaper. In terms of ore tonnage that requires processing, a cut-off grade of 500ppm Sn seems favourable but this advantage may be offset by the costs incurred in selective mining method vs continuous open cast mining which would be ideal at 200ppm Sn cut-off. In addition, the 200ppm Sn cut-off grade could offer favourable life of mine (LoM) due to high ore tonnage of 6.26 million tons ore. Scenario analysis would be recommended during feasibility studies. In the present investigation, additional areas of mineralisation, which were not considered in the evaluation of 1978 have been included, such as areas with previous scattered underground workings.

Considering a cut-off grade of 200ppm and a maximum overburden thickness of 30m, anomalous disseminated tin-bearing ore body within the Lease Granite has an indicated resource of 6.26 million tons of mineralisation averaging 813.05 ppm Sn. The resource would yield 5,085.81 tons of tin which, at present tin price of about US$15,000/ton (December 2009), would appear to be economically viable.

CHAPTER 4: GEOSTATISTICAL EVALUATION

4.1 Introduction

The objective of the study is to estimate the grade, thickness and volume of the deposit given available drill-hole sample data.

4.2 Data checks and analysis

4.2.1 Data checks

Information with Hole/Sample ID, X, Y and Z coordinates, intersections and tin grade for 213 drill-hole samples was received from hard copy files for the Tin project. The sample data was captured in EXCEL and exported from EXCEL spread sheet into MAPINFO. Validation of data was carried out using MapInfo; there were no duplicates. Drill hole RDP042 plotted outside the cluster of all the other samples, however the coordinate was then adjusted to the actual position using old maps from the previous evaluation exercise. The original dataset of 213 drill hole sample is shown in Appendix 1.

There was inconsistency in the detection limit. From the original sample values, twenty-five (25) samples were below detection limit. It is imperative to note that the general detection limit was 120 ppm Sn, however, in some cases, the detection limit varied in different samples, (e.g. 60ppm in RDP054, 125ppm in RDP204 and 1000ppm in RDP002) as shown in Table 4.1**.** In this study, however, all these samples were classified as being below cut-off or detection limit and were assigned a grade value of 0ppm Sn. Thus, 178 samples were above detection limit and had actual grade values.

Table 4.1: Samples below detection limit

Hole ID	RDP001	RDP002	RDP005	RDP007	RDP009	RDP010	RDP012	RDP041	RDP050
Grade(ppm)	<120	**<1000**	**<1000**	**<1000**	**<1000**	**<1000**	**<1000**	<120	<120

Hole ID	RDP053	RDP054	RDP055	RDP056	RDP058	RDP059	RDP062	RDP063	RDP067
Grade(ppm)	<120	<60	<60	<60	<120	<120	<60	<60	<60

Hole ID	RDP073	RDP075	RDP097	RDP099	RDP159	RDP198	RDP204
Grade(ppm)	<120	<120	<120	<120	<60	<120	<125

Ten drill holes were planned but not drilled and these were RDP025, 132, 133, 153, 156, 157, 183, 186, 187 and 199 (Appendix 1). They were not drilled due to a variety of reasons ranging from two holes of different phases plotting on the same position (e.g. RDP025 and RDP002) to holes plotting on the previously worked areas (e.g. RDP 132). Thus, there was a total of 203 drilled holes, see Appendix 1.

It was observed that most samples with overburden of 30m had thin mineralisation which could be too deep for open cast mining (see Figs 3.1 to 3.5 in Chapter 3). Thus 32 samples with overburden of more than 30m were excluded from this study (Appendix 1), with the exception of RDP021A, RDP022A and RDP023 as these were very close to the overburden limit of 30m and with high grade tin values (Table 4.2 and Fig 4.1). The final dataset for this study comprises 171 drill hole samples, and these are shown on base map, Fig 4.2, within the boundary limit. Barren samples within this 30m overburden limit were honoured in this evaluation exercise.

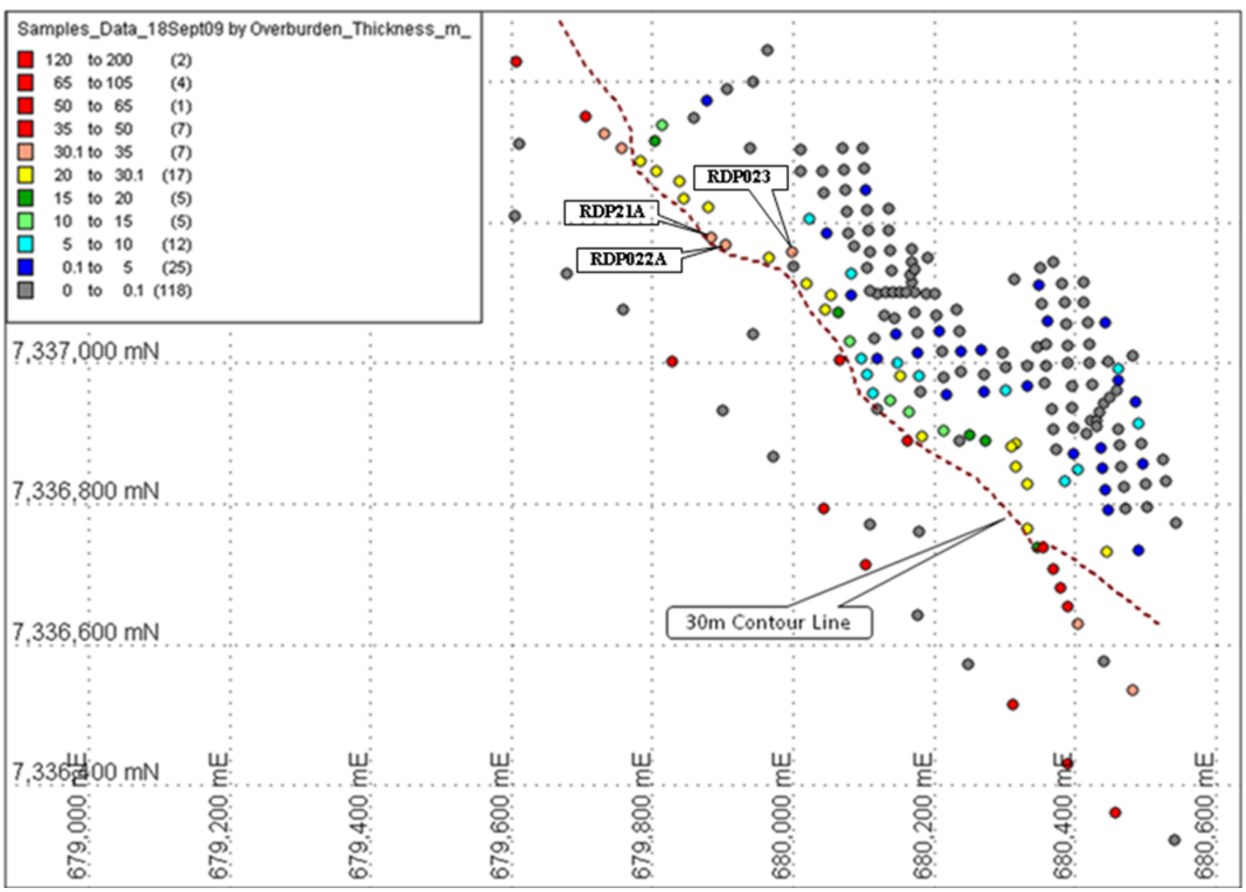

Fig 4.1: Sample data showing 30m contour line

Table 4.2: Mineralisation with greater than 30m overburden

Hole ID	Tin Grade (ppm)	Overburden Thickness(m)	Mineralisation Thickness(m)	Content(m*ppm)	Explanation
RDP001	0	NR	0	0	barren
RDP004	2700	39.98	0.61	1647	too deep
RDP005	0	NR	0	0	barren and too deep
RDP006	6100	40.49	0.61	3721	thin mineralisation, too deep
RDP009	0	NR	0	0	barren and too deep
RDP010	0	NR	0	0	barren and too deep
RDP013	224	40	2	448	thin mineralisation, too deep
RDP014	395	34	5	1975	a bit deep
RDP015	94	31	5	470	low grade
RDP021A	1292	31	21	27132	Close to 30m overburden limit
RDP022A	214	32	4	856	Close to 30m Overburden limit
RDP023	2520	33	6	15120	Close to 30m Overburden limit
RDP044	141	38	15	2115	too deep
RDP045	4223	49	7	29561	too deep
RDP046	163	45	10	1630	too deep
RDP047	209	48	8	1672	too deep
RDP048	844	32	1	844	thin mineralisation, too deep
RDP050	0	NR	0	0	barren
RDP052	380	32	1	380	thin mineralisation
RDP054	0	NR	0	0	barren
RDP055	0	NR	0	0	barren
RDP056	0	NR	0	0	barren
RDP057	451	151	3	1353	too deep
RDP058	0	NR	0	0	barren
RDP059	0	NR	0	0	barren
RDP060	367	102	1	367	too deep, thin mineralisation
RDP061	592	120	1	592	too deep, thin mineralisation
RDP062	0	NR	0	0	barren
RDP063	0	NR	0	0	barren
RDP064	254	83	2	508	too deep, thin mineralisation
RDP065	644	65	5	3220	too deep, pipe possibility
RDP066	200	90	15	3000	too deep, pipe possibility
RDP067	0	NR	0	0	barren
RDP075	0	NR	0	0	barren
RDP076	257	64.6	5.2	1336.4	barren

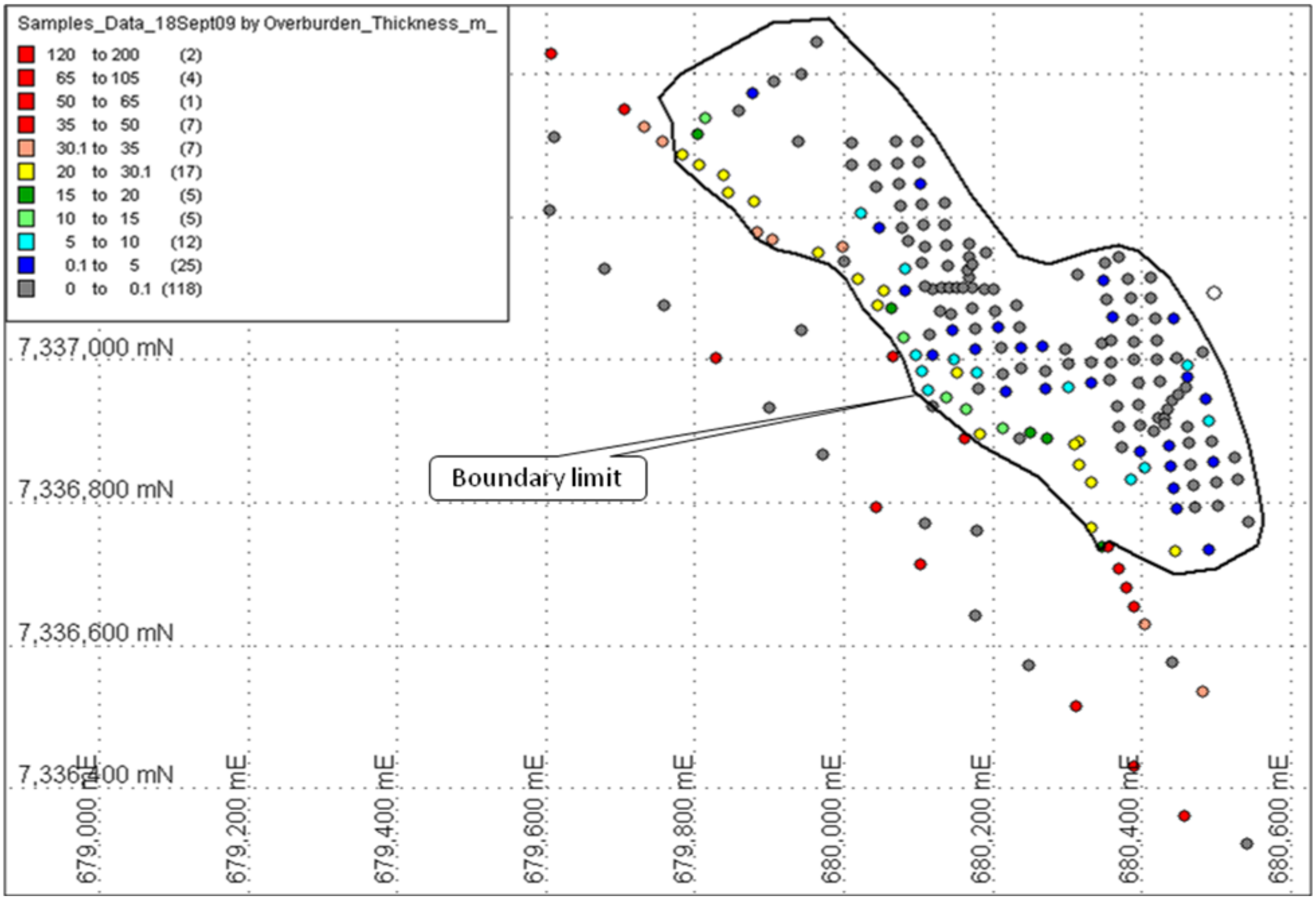

Fig 4.2: Sample data showing the boundary and the area under study

It should further be noted that where tin grade is below the detection limit, the overburden thickness cannot be recorded and MapInfo automatically assumes that the overburden is zero, which is not necessarily true, e.g. the grey samples outside the boundary in Fig 4.2 do not reflect zero overburden since in that region, overburden is more than 30m. This is also shown in Table 4.3 and Fig 4.3. The final 171 samples were re-imported into MapInfo.

Table 4.3: Samples that are below detection limit and are within the boundary limit

Hole ID	Tin Grade (ppm)
RDP002	<1000
RDP007	<1000
RDP012	<1000
RDP041	<120
RDP053	<120
RDP073	<120
RDP097	<120
RDP099	<120
RDP159	<60
RDP198	<120
RDP204	<125

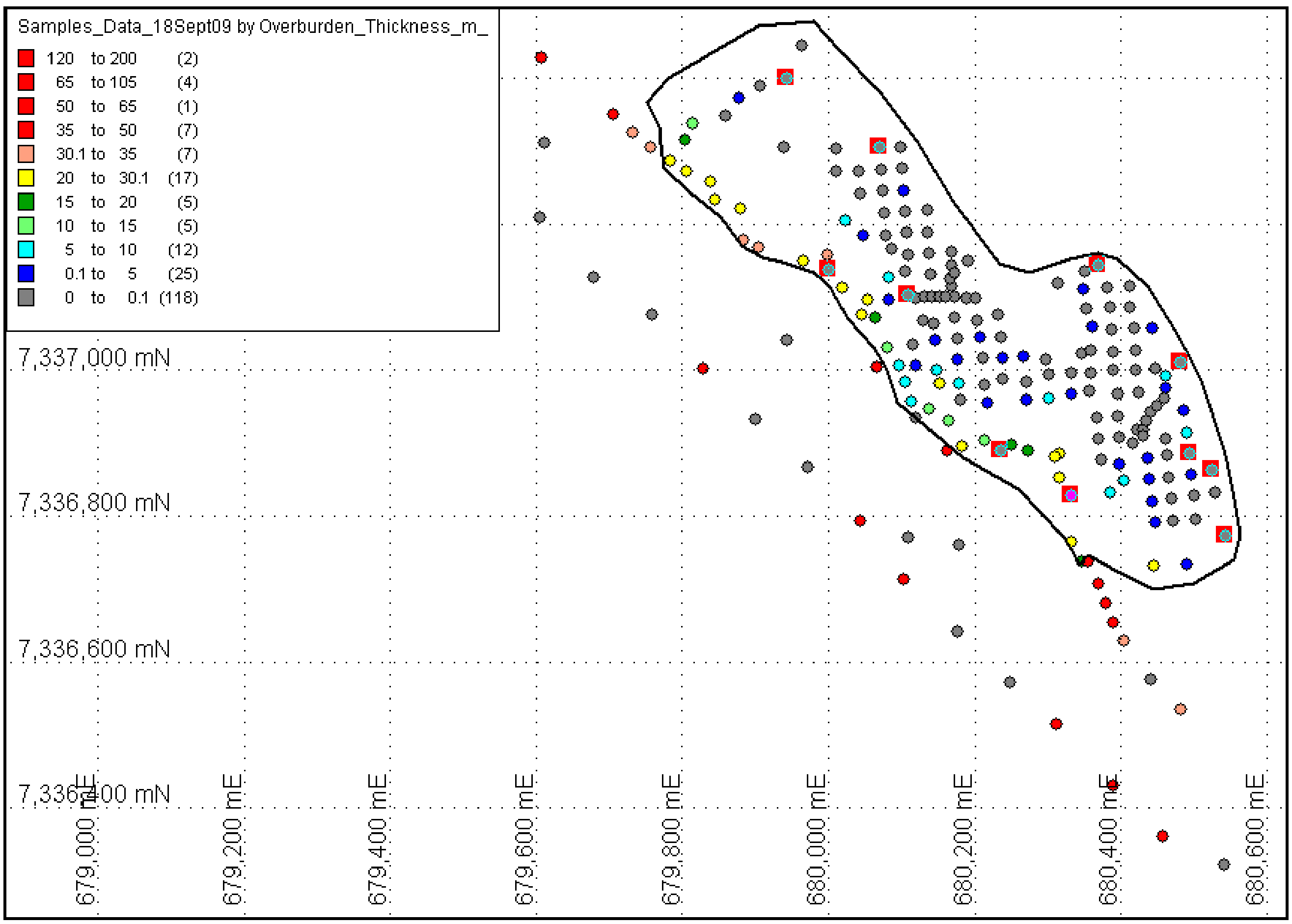

Fig 4.3: Highlighted samples are below detection limit within the area under study

4.2.2 Data analysis

Three more variables were calculated: mineralisation thickness (m), overburden thickness (m) and tin content (m*ppm). Mineralisation and overburden thickness were both computed from the intersections. Tin content was calculated from the product of grade and mineralisation thickness. The three variables: grade (ppm), mineralisation thickness (m), overburden thickness (m), and area (m^2) were analysed in detail and used for estimation. The tin grade data is lognormally distributed since the majority (96%) of the samples have low tin values (Minnitt, 2004). Only six samples have high grade values of greater than 2,000ppm Sn (Fig 4.7). However, the data was treated using conventional methods for a normally distributed dataset after masking the very high grade values (>5,000ppm Sn). The density for the Lease Granite was assumed to be 2.57t/m^3.

4.3 Basic Statistics

The samples were collected continuously from both drill core and drill chips at 1m interval. Most samples from the outcrop mineralisation were collected from percussion drill chips. The drill hole spacing was generally 35m by 35m. The sample data with the following variables was received: sample IDs, location (X ,Y and Z coordinates), grade and mineralisation intersections. As stated previously necessary additional variables were calculated from the raw data i.e. grade (ppm), mineralisation thickness (m) and overburden thickness (m). Appendix 1 is a list of the original 213 boreholes/samples. Summary statistics were performed on both 203 original drilled samples, (Table 4.4a), on bounded 171 samples (Table 4.4b) and on final cleaned dataset of 169 samples (Table 4.4c).Two very high grade (>5,000 ppm Sn) samples were masked. The advantage of masking the two high grade samples can be observed on the tables of summary statistics, Tables 4.4b and 4.4c, although the means are fairly comparable (639.68 and 580.85) the difference between the mean and the maximum for the 171 samples is much higher (639.68 and 6,114 ppm) than that of 169 samples, (580.85and 3019ppm).

Consequently, the 169 samples constitute the final dataset that was used for the variogram modelling for grade estimation purposes in this study. Summary statistics of “cleaned” data for the Lease Granite is shown in Table 4.4c. For kriging purposes, the total number of acceptable samples is 171 and 158 are positive and therefore 11 (or 7%) are barren samples (below cut-off).

In the case of all 203 drilled samples, a maximum of 6,114ppm Sn, with an average grade of 628.7ppm Sn was obtained. For the bounded 171 samples, the maximum grade value is 6,114ppm with an average grade of 639.68 ppm Sn. Thus the removal of 32 samples did not have a big impact on the average grade. However, removal of two very high grade samples have greatly altered the statistics of the mineral resource, a maximum grade of 3,019 ppm Sn with an average of 580.85 pp Sn, shows that the two high grade samples had a strong influence on the statistics of the other samples.

The coefficient of variation for grade reduces from 1.17 in 171 samples to 0.89 in 169 samples. The average mineralisation thickness for the 169 samples is 12.64m with a maximum thickness of 30m, overburden has an average thickness of 4.85m with a maximum thickness of 33m, skewness and kurtosis have greatly improved by the removal of the two high grade samples from 4.06 and 23.29 to 1.85 and 4.17 respectively, (Table 4.4c).

Table 4.4a: Summary sample statistics of 203 drilled samples (After 10 boreholes planned but not drilled holes were removed)

	Tin Grade(ppm)	**Overburden Thickness(m)**	**Mineralisation Thickness(m)**	**Content(m*ppm)**
Count	203	179	202	203
Min	0	0	0	0
Max	6114	151	30	122,280
Mean	628.68	10.57	11.13	8,622.78
Std.Dev	862.56	21.78	8.18	14,815.7
Var	744014.95	474.47	66.86	219,504,959.6
Skewness	3.94	3.45	0.22	4.72
Kurtosis	19.7	14.99	-1.12	29.48
CoV	1.37	2.06	0.73	1.72

Table 4.4b: Summary sample statistics of 171 samples with overburden of less or equal to 33m (after removal of 32 samples)

	Tin Grade(ppm)	Overburden Thickness(m)	Mineralisation Thickness(m)	Content(m*ppm)
Count	171	160	171	171
Min	0	0	0	0
Max	6114	33	30	122,280.00
Mean	639.68	4.79	12.64	9,915.70
Std.Dev	750.63	8.76	7.81	15,660.25
Var	563,440.89	76.77	60.99	245,243,343.1
Skewness	4.06	1.8	0.05	4.51
Kurtosis	23.29	1.88	-1.06	26.48
CoV	1.17	1.83	0.62	1.58

Table 4.4c: Summary sample statistics of 169 samples with overburden of less or equal to 33m (after removal of 2 high grade samples)

	Tin Grade(ppm)	Overburden Thickness(m)	Mineralisation Thickness(m)	Content(m*ppm)
Count	169	158	169	169
Min	0	0	0	0
Max	3019	33	30	72,456.00
Mean	580.85	4.85	12.54	8,644.68
Std.Dev	519.13	8.8	7.8	10,436.6
Var	269,493.59	77.45	60.86	108,922,669.9
Skewness	1.85	1.78	0.08	2.59
Kurtosis	4.17	1.81	-1.04	9.59
CoV	0.89	1.81	0.62	1.21

4.4 Scatter plots

Scatter plots (Fig 4.5) were performed to establish the correlation between a number of variables. The correlation coefficients are shown in Table 4.5.

Table 4.5: Correlation coefficients for sample variables

Variable scatter plot	Correlation coefficient
Tin Grade (ppm) vs Mineralisation Thickness (m)	0.1141
Tin Grade (ppm) vs Tin Content (m*ppm)	0.6407
Tin Grade (ppm) vs Overburden Thickness (m	0.0007
Mineralisation Thickness (m) vs Tin Content (m*ppm)	0.4096
Mineralisation Thickness (m) vs Overburden Thickness (m)	0.0073
Tin Content (m*ppm)vs Overburden Thickness (m)	0.0284

There is very poor correlation between overburden thickness and the other variables, for instance the correlation coefficient of overburden thickness and tin grade, is 0.0007. A weak correlation (rho=0.1141) is also observed between grade and mineralisation thickness. A fairly good correlation exists between grade and tin content (rho=0.6407) and also between mineralisation thickness and tin content (rho=0.4096), as one would have expected since tin content is derived from product of grade and mineralisation thickness. The poor correlation between overburden thickness and other variables is also expected as the variables are independent of each other in terms of genesis and occurrence. A weak correlation between tin grade and mineralisation thickness would imply that thicker mineralised zones would have a bit higher grades and this has been observed in the contour maps in Chapter 2.

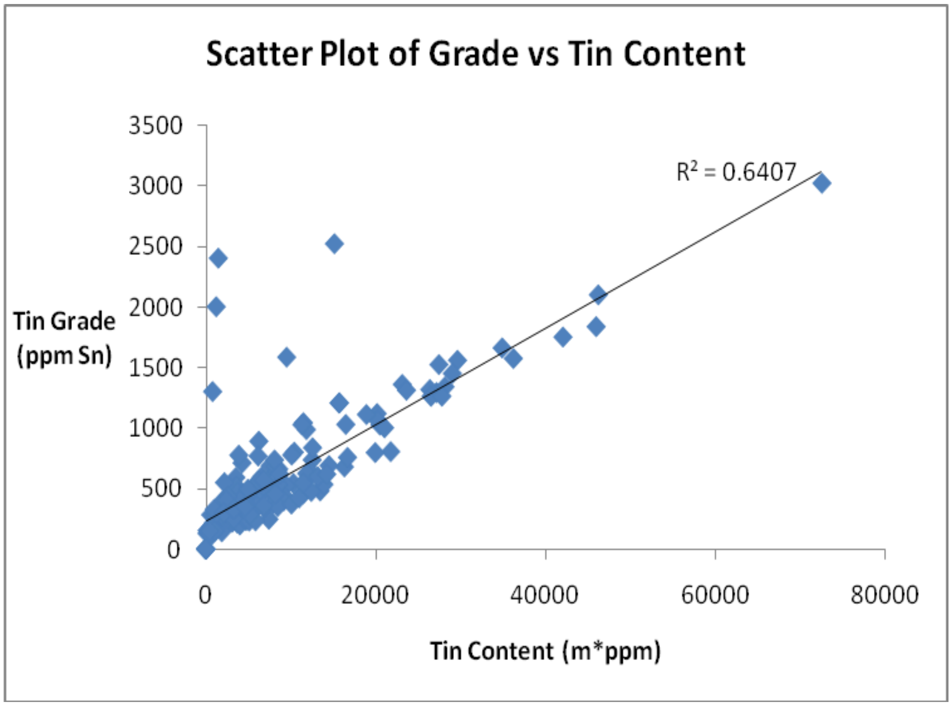

Fig 4.5a: Scatter plot showing correlation between tin grade and tin content

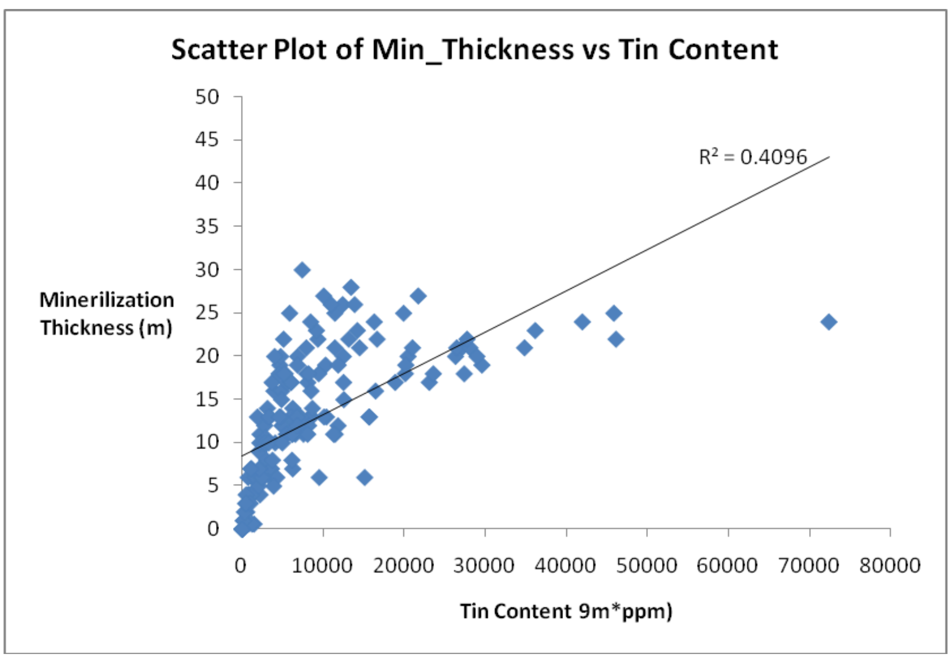

Fig 4.5b: Scatter plot showing correlation between mineralisation thickness and tin content

4.5. Variography

4.5.1 Grade

Sample grade data (ppm Sn) is shown in Fig 4.6, where the colour of the diamond symbol is related to the grade value. The high grade values are more concentrated in the northern central (around RDP080) and southern central (around RDP146) portions as shown in the Fig 4.6.

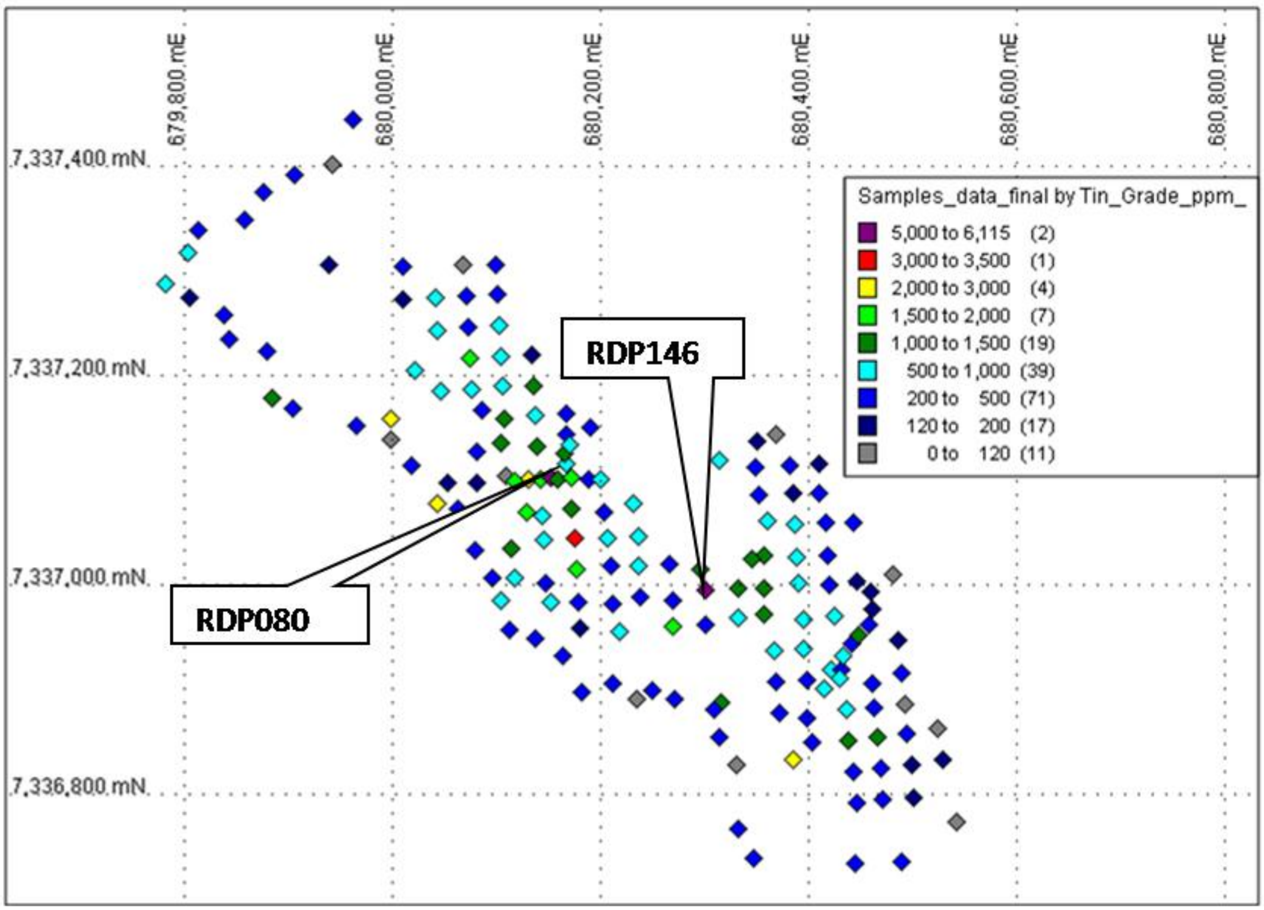

Fig 4.6: Base map showing high grade values in the northern cental (RDP080) and southern central (RDP146) portion

Figure 4.7 shows the histograms of sample data before (above) and after (bottom) removal of the high grade values of 5,107ppm Sn and 6,114ppm Sn. The high grade values were removed and the remaining dataset of 169 samples was used for variogram modelling. Figure 4.8 shows a Grade Variogram Map displaying isotropy in the mineralised Lease Granite. The anisotropy was tested in different directions but the variograms produced were not convincing. As a result, the omni-directional semi-variogram model was utilised for modeling the spatial structure.

The variogram grade model in Fig 4.8 has a range of 147m, a nugget effect of 82,602 with a total sill of 222,390. The nugget effect is about 37% of the total sill indicating a fairly low random component in this grade model. The lag distance of 40m was chosen because it is more or less the sample spacing distance.

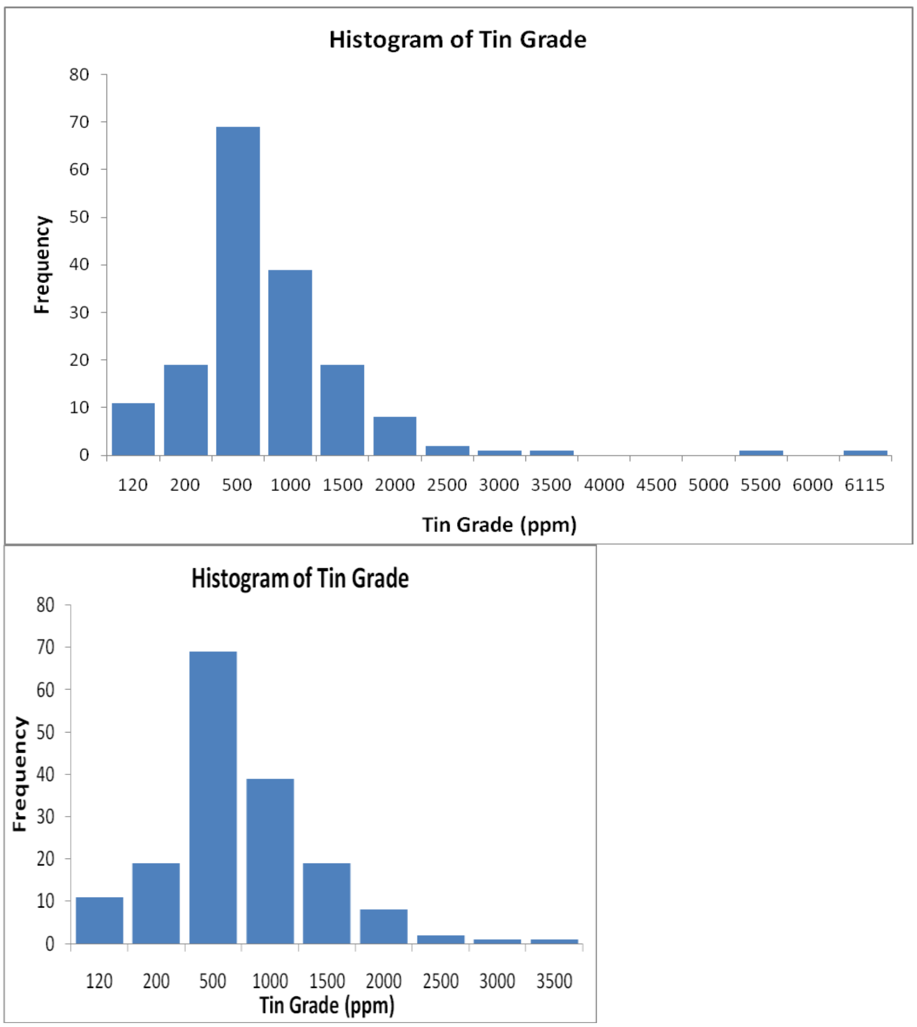

Figure 4.7: Grade histogram for all 171 samples (top) and grade histogram with excluded two high grade sample above 5000ppm (bottom)

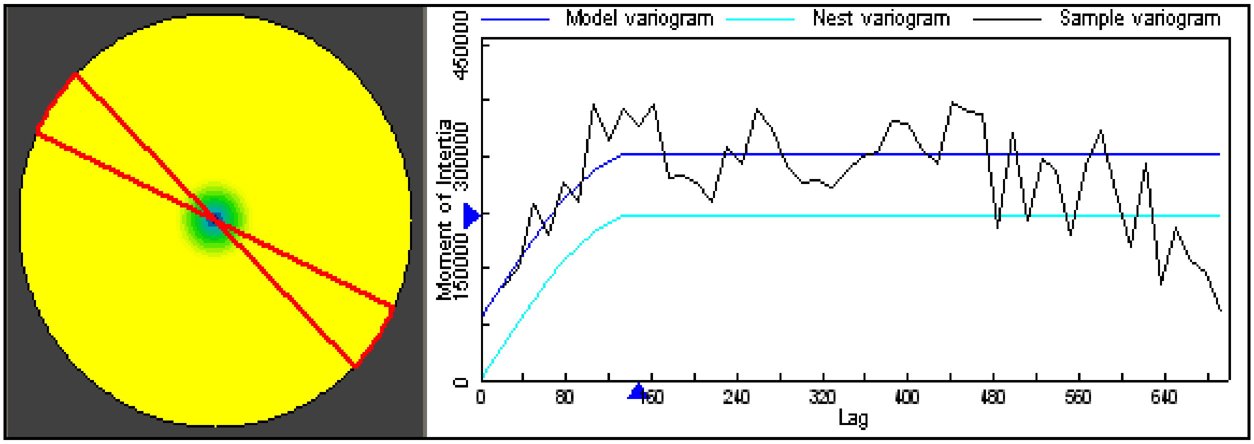

Fig 4.8: Variogram map and a semi-variogram model of tin grade in mineralised Lease Granite

4.5.2 Mineralization thickness

Fig 4.9 is a histogram showing mineralised Lease Granite thickness. The average thickness is 12.64m with a maximum of 30.0m. However, Fig 4.10 shows that thicker samples are present in the central, south-western region of the SE-NW elongate structure of the mineralisation while thinner mineralisation is dominant on the general periphery of the elongate structure**.**

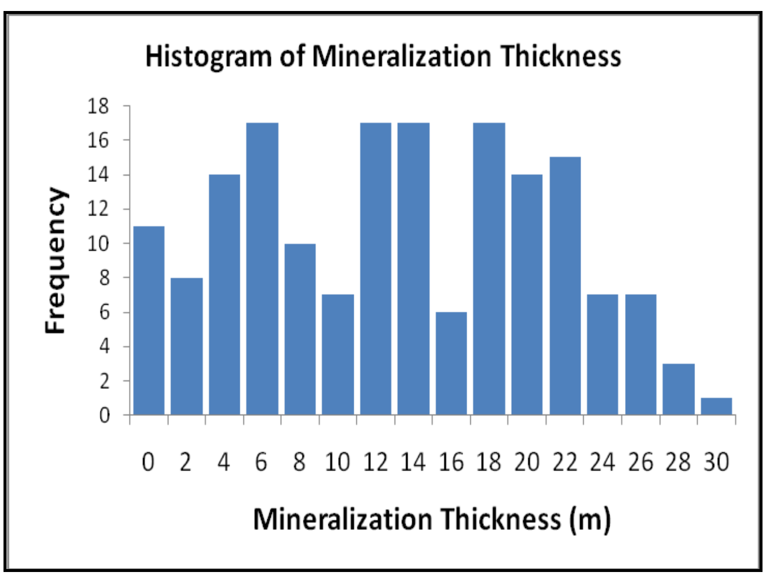

Fig 4.9: Mineralisation thickness histogram

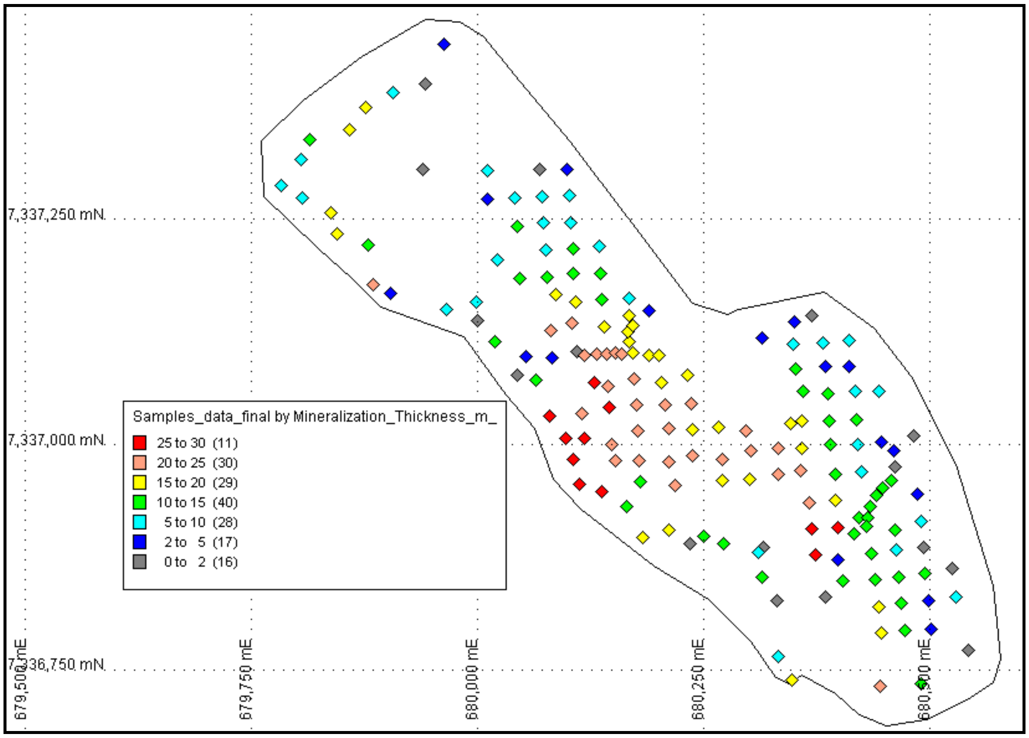

Fig 4.10: Mineralisation thickness data for Lease Granite.

The mineralised Lease granite was tested for anisotropy (Fig 4.11) and there was no evidence of its presence and therefore an omni-directional variogram model was chosen and is shown in Fig 4.11. The spherical range is 257.47m, nugget effect is 1.46 and the total sill is 64.4 and the nugget effect is about 2.2% of the total sill showing a very low random component in this mineralization thickness model. The lag distance is 40m which is more or less equal to sample spacing.

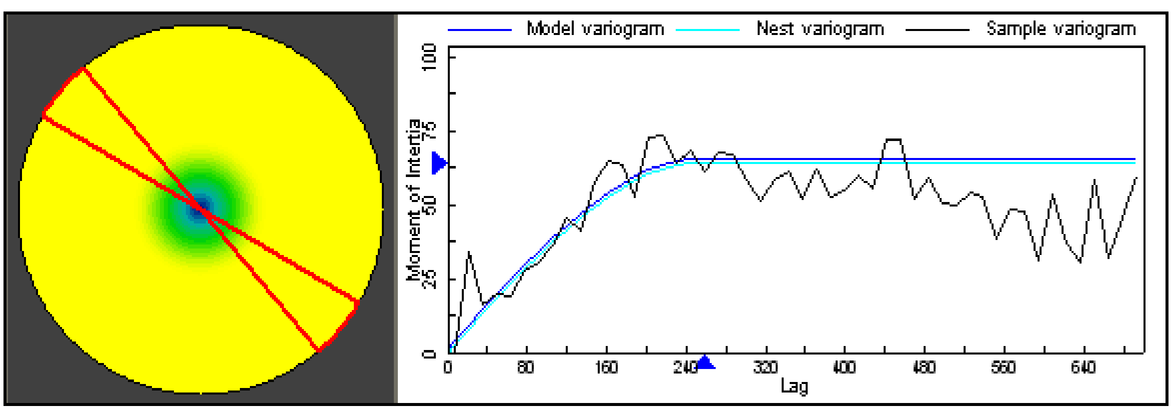

Fig 4.11: Variogram map and a semi-variogram model for the thickness of mineralised Lease Granite

4.6 Neighbourhood optimisation

4.6.1 Selection of Block Size

Generally, block size should not be less than half the sample spacing. A block size greater than sample spacing would be ideal if samples are too few or if there are some doubts about the sample integrity. Small blocks (smaller than sample spacing) are ideal for high resolution e.g. for mine planning. In estimation, the method of sample block centering generally gives more reliable results than the one with samples at the corners of the block. In this study a 35m x35m block size was chosen because it is more or less equal to the spacing of the drill holes that were drilled to test the mineralisation of the outcropping Lease Granite. As a result, a grid of 35m x 35m was created. The blocks were further discretised or decimated into smaller sub-blocks of 5m by 5m.

4.6.2 Neighbourhood optimisation

During selection of neighbourhoods, it was noted that the higher the neighbourhood radius, the lower the variance and the higher the slope of regression and the total percentage of negative weights should ideally be zero. An optimum neighbourhood was chosen with the lowest variance, highest slope of regression and with very minimal (<1%) total percentage of negative weights. The optimum neighbourhood radius for grade and mineralisation thickness was 80m. Due to the outline shape of the deposit, the kriged blocks could not cover the whole resource area and sparsely sampled areas were estimated using a larger neighbourhood radius. The neighbourhood radii for grade and thickness for full coverage of the deposit were 100m each.

4.7 Kriging

The variogram models and the neighbourhoods outlined earlier were used to produce block estimates for grade (ppm Sn) and mineralisation thickness (m). The kriged block estimates are lower but they compare well with the averages derived from the samples, the estimates fall within 20% of the sample average. The estimates for grade and thickness are shown in Figs 4.12 and 4.14 respectively and are summarised in Table 4.6.

Table 4.6: Sample means and Kriged estimates for grade and thickness

Variable	Sample Mean	Kriged Block Estimate (calculated)
Grade(ppm)	580.85	493.03
Thickness(m)	12.64	10.48

4.7.1 Grade

The block estimate for grade (Fig 4.12) generally shows a high grade zone in the central western portion of the elongate structure. The average block grade is 493.03 ppm Sn with a maximum grade of 1,156.44 ppm Sn. The block estimate grade is relatively close to the mean grade of sampling data of 580.85 ppm Sn, being 15% lower than the sample mean.

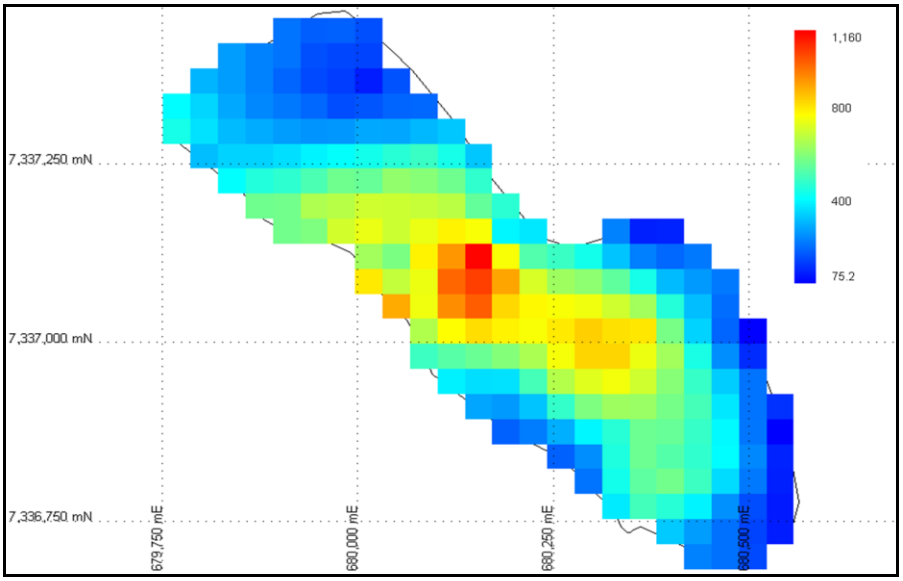

Fig 4.12: Final grade (ppm Sn) block estimate of mineralised Lease Granite

The histogram of estimated grade for mineralised Lease Granite is shown in Fig 4.14 and shows a mean of 493.03m and a maximum of 1156.44 as compared to the sample mean of 580.85m with a maximum of 3,019 ppm Sn. The block estimate falls within 15% of the sample mean grade.

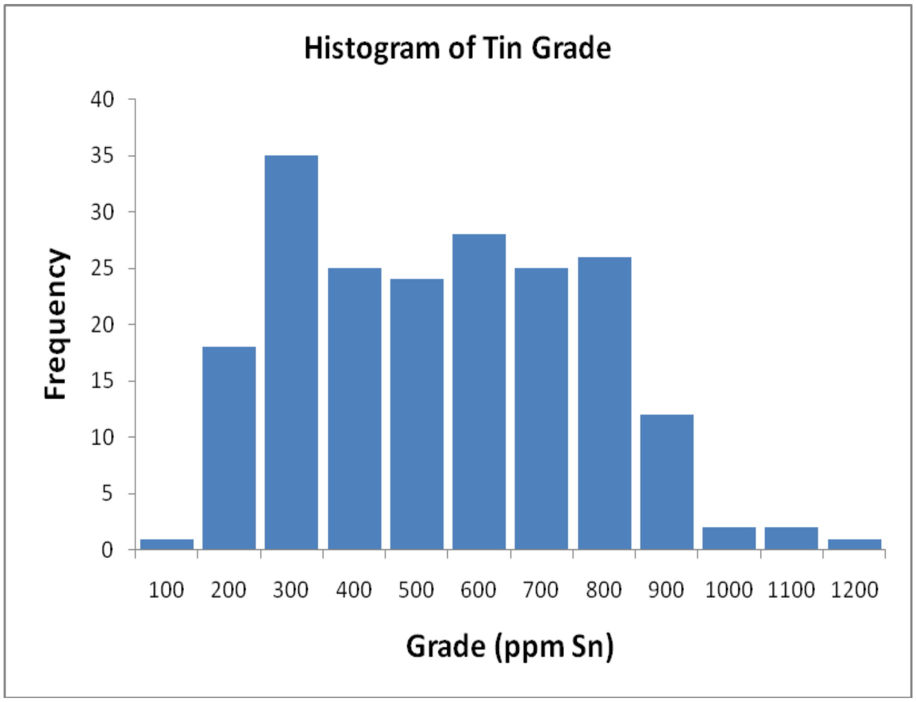

Fig 4.13: Histogram of estimated grade values for mineralised Lease Granite

4.7.2 Mineralisation thickness

The block estimate for mineralised Lease Granite thickness (Fig 4.14) generally shows a thicker central zone trending SW-NE and thicker central western portion of the elongate shape with generally thinner margins and this is in line with the shape of the domed lenticular mineralisation described earlier in Chapters 2 and 3. The maximum block mineralisation thickness estimate is 26.1m with an average kriged estimate of 10.48m which is 17% lower that the sample mean of 12.64m.

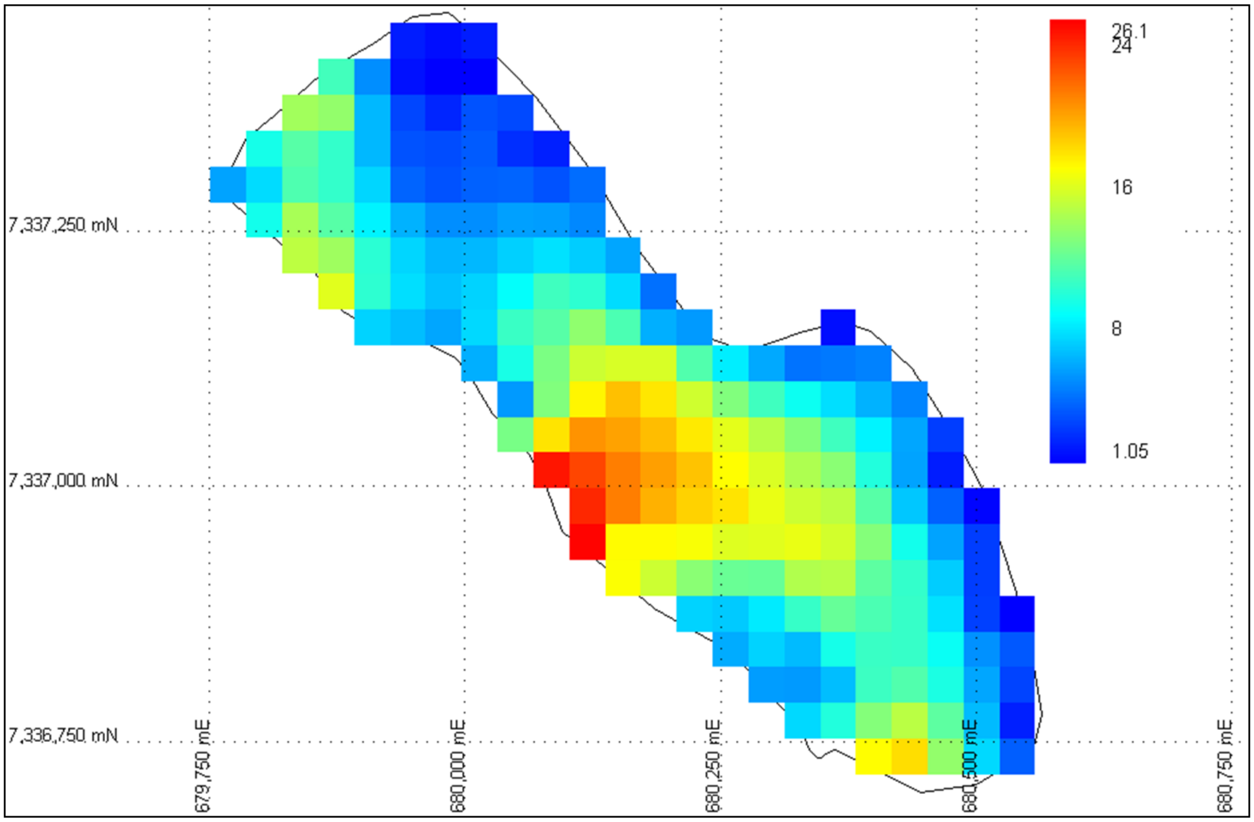

Fig 4.14: Block estimates for Lease Granite thickness (m)

The histogram of estimated thickness for mineralised Lease Granite is shown in Fig 4.15 and shows a mean of 10.48m and a maximum of 26.07 as compared to the sample mean of 12.64m with a maximum of 30m.The block estimate falls within 20% of the sample mean thickness.

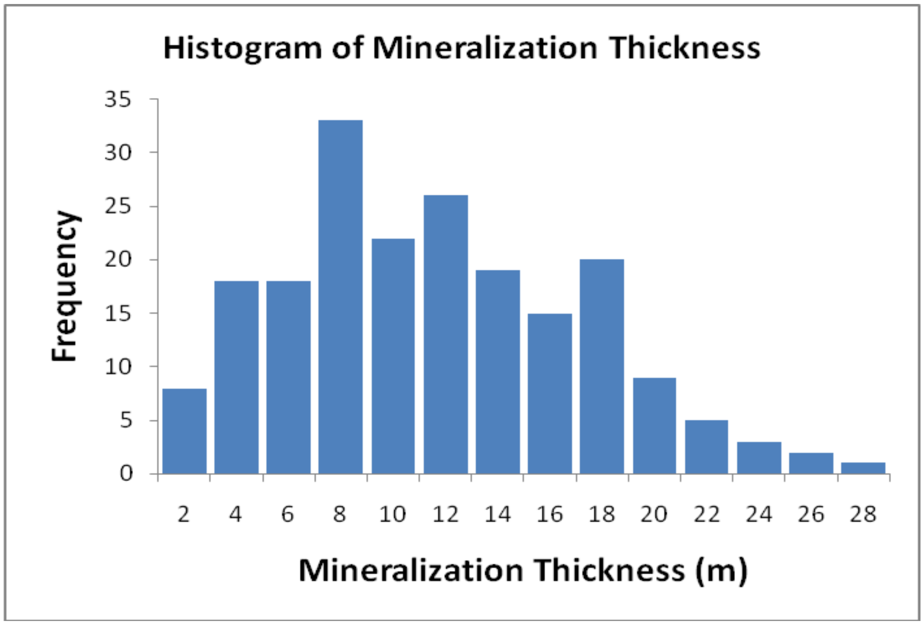

Fig 4.15: Histogram of estimated thickness values for mineralised Lease Granite

4.8 Estimation validation

Data from block estimates was checked and the volume was calculated by finding the product of area (m^2) and mineralisation thickness (m). A block listing is presented in Appendix 3. In terms of summary mineral resource, the average thickness was computed from dividing total volume (m^3) by total area (m^2), and the result is 10.48m. The calculations were undertaken in Excel spreadsheets (Appendix 3). The summary mineral resource for grade and mineralisation thickness is shown in Table 4.7. The block estimates for these variables have been compared with the sample means as described earlier in Table 4.6. These results are fairly close to those of the sample values, although slightly lower.

Table 4.7: Summary statistics of grade and thickness block estimates for the mineralised Lease Granite.

	Mineralisation Thickness(m)	Grade(ppm)	Area(m^2)	Volume(m^3)
Sum	2,085.7	98,113.5	243,775	2,555,018
Count	199	199	199	199
Max	26.1	1,156.4	1,225	31,940
Min	1.1	75.2	1,225	1,291
Range	25.0	1,081.2		30,649
Mean	10.5	493.0	1,225	12,839.3
Var	31.97	51,909.10		47,980,334.8
StDev	5.65	227.84		6926.78
Cov	0.54	0.46		0.54
Block Estimate; Mineralization Thickness (m) = 2,555,018.34/243,775 = **10.48**				
Block Estimate; Grane (ppm Sn) = **493.03**				

Table 4.8: Summary statistics of grade and thickness block estimates for the mineralised Lease Granite at 200ppm Sn cut-off.

	Mineralisation Thickness(m)	Grade(ppm)	Area(m^2)	Volume(m^3)
Sum	2,026.20	95,163.35	220,500.00	2,482,099.16
Count	180.00	180.00	180.00	180.00
Max	26.07	1,156.44	1,225.00	31,939.75
Min	1.58	212.35	1,225.00	1,938.43
Range	24.49	944.08	0.00	30,001.32
Mean	11.26	528.69	1,225.00	13,789.44
Var	28.64	43,897.23	0.00	42,976,011.30
StDev	5.35	209.52	0.00	6555.61
Cov	0.48	0.40	0.00	0.48

4.9 Mineral resource statement

Estimated grade for the mineralised Lease Granite is 493.03 ppm Sn compared to sample mean of 580.85 ppm Sn. The estimated average thickness is 10.48m as compared to 12.64m derived from sample data. The estimates are comparable with the averages derived from the samples (i.e. within 17% of the sample means).

Table 4.9: Mineral Resource Statement

Block Estimate	Total Resource		200ppm Cut off	
	Resource	5% Discounted	Resource	5% Discounted
Total Volume	2,555,018	2,427,267	2,482,099	2,357,994
Density of Lease Granite	2,570.00	2,570.00	2,570.00	2,570.00
Ore kg	6,566,397,124	6,238,077,268	6,378,994,841	6,060,045,099
Ore Ton	6,566,397	6,238,077	6,378,995	6,060,045
Grade	493.03	493.03	528.69	528.69
Tin grammes	3,237,449,739	3,075,577,252	3,372,480,569	3,203,856,541
Tin Kg	3,237,449.74	3,075,577.25	3,372,480.57	3,203,856.54
Tin Ton	3,237.45	3,075.58	3,372.48	3,203.86

The density for the mineralised Lease Granite is assumed to be 2.57 t/m^3. When the resource tonnage is discounted at 5% for the provision of volume consumed in the previous underground workings, the Lease Granite mineralisation can be classified as an indicated resource (SAMREC code, 2009) with 3,075.58 tons of tin in 6.24 million tons of mineralised Lease Granite at an average of 493.03 ppm Sn (Table 4.9). Considering a cut-off grade of 200ppm and overburden limit of 30m the resource tonnage would be 6.06 tons at an average grade of 528.69 ppm Sn and would contain 3,203.86 tons of tin (Table 4.9). The average mineralisation thickness would be 11.26m (Table 4.8).

4.10 Kriged and polygonal estimates at 200ppm Sn cut-off

The 200ppm Sn cut-off polygonal grade estimate is 813ppm Sn while that of kriged block estimate is 528.69 ppm Sn. The resource tonnage is 6.26 million tons for polygonal estimate while that of kriged block estimation is 6.06 million tons, when 5% is discounted for underground workings. The tin tonnage is 5,085.81 tons for the polygonal estimate whereas that of kriged block estimate is 3,203.86 tons (Table 4.10).

The resource tonnages for kriged and polygonal estimates are quite comparable, about 6 million tonnes. However, the kriged grade estimate is lower than the polygonal estimate. This difference is due to the fact that the few very high grade samples were masked i.e. they were not used in the construction of the variogram model. In polygonal estimation, the few high grade samples were not masked and hence the higher average grade. The tin tonnage for kriged estimate, (3,203.86 tons Sn) is lower than for the polygonal estimate (5,085.81 tons Sn) due to differences in average grades.

Table 4.10: Comparison between polygonal and kriged block estimates.

Estimate	Polygonal Estimate: 200ppm Sn Cut-off		Block Estimate : 200ppm Sn Cut off	
	Resource	5% Discounted	Resource	5% Discounted
Total Volume	2,562,037	2,433,935	2,482,099	2,357,994
Density of Lease Granite	2,570.00	2,570.00	2,570.00	2,570.00
Ore kg	6,584,435,251	6,255,213,488	6,378,994,841	6,060,045,099
Ore Ton	6,584,435	6,255,213	6,378,995	6,060,045
Grade	813.05	813.05	528.69	528.69
Tin grammes	5,353,489,413	5,085,814,943	3,372,480,569	3,203,856,541
Tin Kg	5,353,489	5,085,815	3,372,481	3,203,857
Tin Ton	5,353.49	5,085.81	3,372.48	3,203.86

CHAPTER 5: MINERAL EXTRACTION AND RECOVERABILITY

5.0 Introduction

Cassiterite has a high relative density in relation to its gangue constituents and thus makes it ideal for the application of gravity separation techniques. On the other hand, its relative hardness makes it extremely brittle. The cassiterite's brittleness must be taken into consideration during crushing and grinding operations before concentration. Thus, cassiterite grains should be recovered at the earliest possible stage and at their largest size. The efficiency of gravity – concentration processes decreases markedly once the size of the particles is reduced to below about 43μm, (Falcon, 1982).

Two methods of cassiterite recovery exist; those that apply to soft rock (alluvial/eluvial) deposits and those that apply to hard rock (mineralisation in parent in situ rock) deposits. As highlighted earlier in Chapter 1, tin mineralisation in the Roodepoort-Groenfontein project area is confined to the Lease Granite and thus hard rock deposit recovery techniques are applicable. Thus, conventional crushing and grinding methods can be used to liberate cassiterite from associated gangue materials. Due to its extreme brittleness, however, significant amounts of very fine particles can be produced resulting in losses of tin in succeeding processing stages. Froth floatation is used to upgrade particles less than 43μm but cannot treat particles less than 6μm in size. The less than 6μm particles can account for about 6% of the metallic tin entering the plant (Falcon, 1982). Different combinations of same crushing and milling equipment, followed by a wide variety of gravity-concentration devices for further beneficiation are used in most hard rock tin concentrators.

A plant similar to the one used to recover tin from tailings dumps at Zaaiplaats Tin Mines could be used to process ore from the deposit, taking care not to over grind the ore material. Crushing of the run-of-mine ore to a size suitable for first stage of gravity separation (about 12mm) is therefore important. The stages after this would thus be as follows (Zelda Real Estate, 2006):

- Grinding in a rod mill
- Spiral and Mozely Table to scavenge the fine particles
- Smelting using full-output's Melt Reduction technology to further increase value of tin content

A recovery of 70% tin should be achieved. A plant, based on the above specifications could treat 100-150 tons /hour, say 3,600 tons/day or 1,000,000 tonnes/yr. Plant operating cost can be in the order of US$1,270/ton Sn (Zelda Real Estate, 2006).

Discounted cash flow (DCF) analysis to verify the economic viability of the deposit will be considered in the next chapter where mining, metallurgical, marketing, legal, environmental, social, governmental and economic considerations will be assessed to convert the resource into a reserve.

5.1 Mintek approach: proposed pilot plant scoping study

It is proposed that MINTEK should conduct a laboratory scale, scoping gravity separation testwork on run of mine (ROM) material with an average head grade of 800ppm (0.080% Sn) at a cut-off grade of 200ppm Sn (0.02% Sn).

Another scenario would be to consider conducting a laboratory scale, scoping gravity separation testwork with a run of mine (ROM) material with average head grade of 1,288ppm (0.1288% Sn) at a cut-off grade of 500ppm Sn (0.05% Sn).

5.1.1 Objectives

An optimum method to extract or recover tin from Lease Granite should be sought. To achieve this it would be necessary to determine:

- appropriate processing route to obtain a saleable Sn product.
- possible recovery of other saleable by-products such as tungsten, molybdenum, fluorspar, etc.

5.1.2 Procedures

5.1.2.1 Upfront mineralogical evaluation

Mineralogical investigation and evaluation would be useful in:

- determining the liberation size of the Sn minerals.
- carrying out a size-by-assay analysis on the ROM feed material.

Based on the mineralogical findings and sizing results a decision will be made on the most feasible processing route. Two options have been proposed, option A for coarse and option B for fine processing (Mintek proposal report, 2008).

5.1.2.2 Processing options

Option A: Coarse processing

Procedure
The processing of coarse material would require the following steps:

1. Crushing material to -12mm
2. Heavy liquid separation(HLS) testwork on -12+1mm fraction
3. Shaking table testwork on -1mm fraction
4. Teeter-bed separation(TBS) separation testwork on -1mm fraction
5. Falcon testwork on -1mm fraction

Option B: Fine processing
Procedure
The processing of fine material would require the following steps:

1. Crushing material to -1mm
2. Milling to a grind determined by mineralogical evaluation
3. Shaking table testwork on the milled fraction
4. Teeter-bed separation(TBS) testwork on the milled fraction
5. Falcon testwork on the milled fraction

The flowsheet below is a proposal from MINTEK for the processing of tin ROM material for both Option A and B (Fig 5.1).

MINTEK expects to receive approximately 2.5 tons of -30mm Sn ROM material. The material will be weighed, blended and split into the following representative sub-samples for evaluation purposes:

1. 200kg representative sub-sample for chemical head analysis and mineralogical evaluation
2. 500kg representative sub-sample for size-by-assay analysis

The remaining material would be prepared for both Option A and Option B processing testwork, as per the procedure outlined in detail in the MINTEK unofficial proposal report,(Mintek proposal report, 2008).

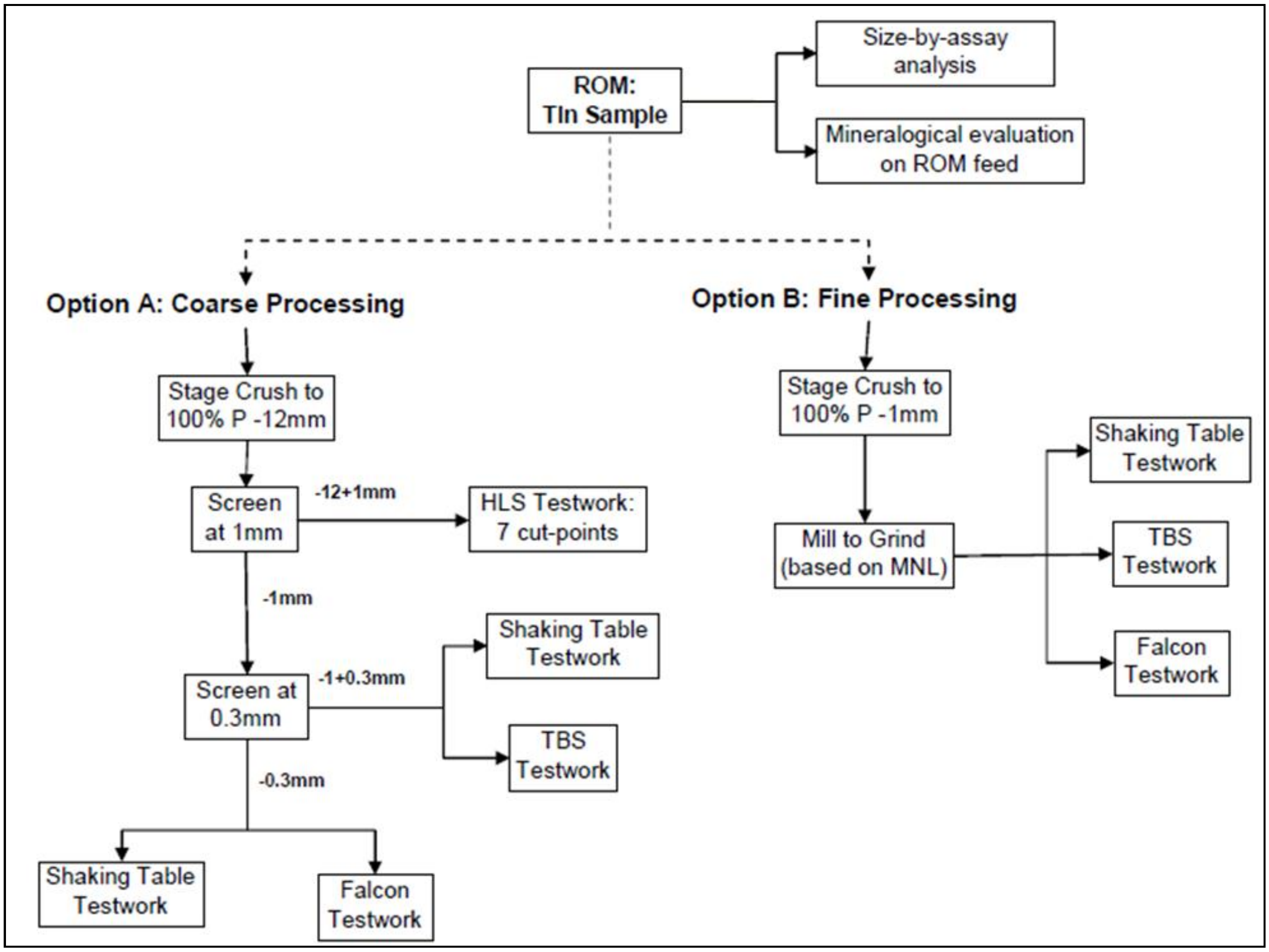

Fig 5.1: Illustration of the proposed test-work program for both Option A and Option B, (after MINTEK, Oct 2008)

Deliverables

Mintek planned to carry out test-work on Lease Granite samples to determine its metallurgical properties and the following would be the deliverables:

- Results of test work
- Final comprehensive report providing details of the test procedure adopted and an interpretation of the results
- MINTEK will return all remaining samples

5.2 Valchem approach: Plant design

Valchem proposed the following conceptual design for the hard rock tin recovery plant, (Valchem, 2008). The accompanying proposed budget for such a plant is tabulated in Table 5.1.

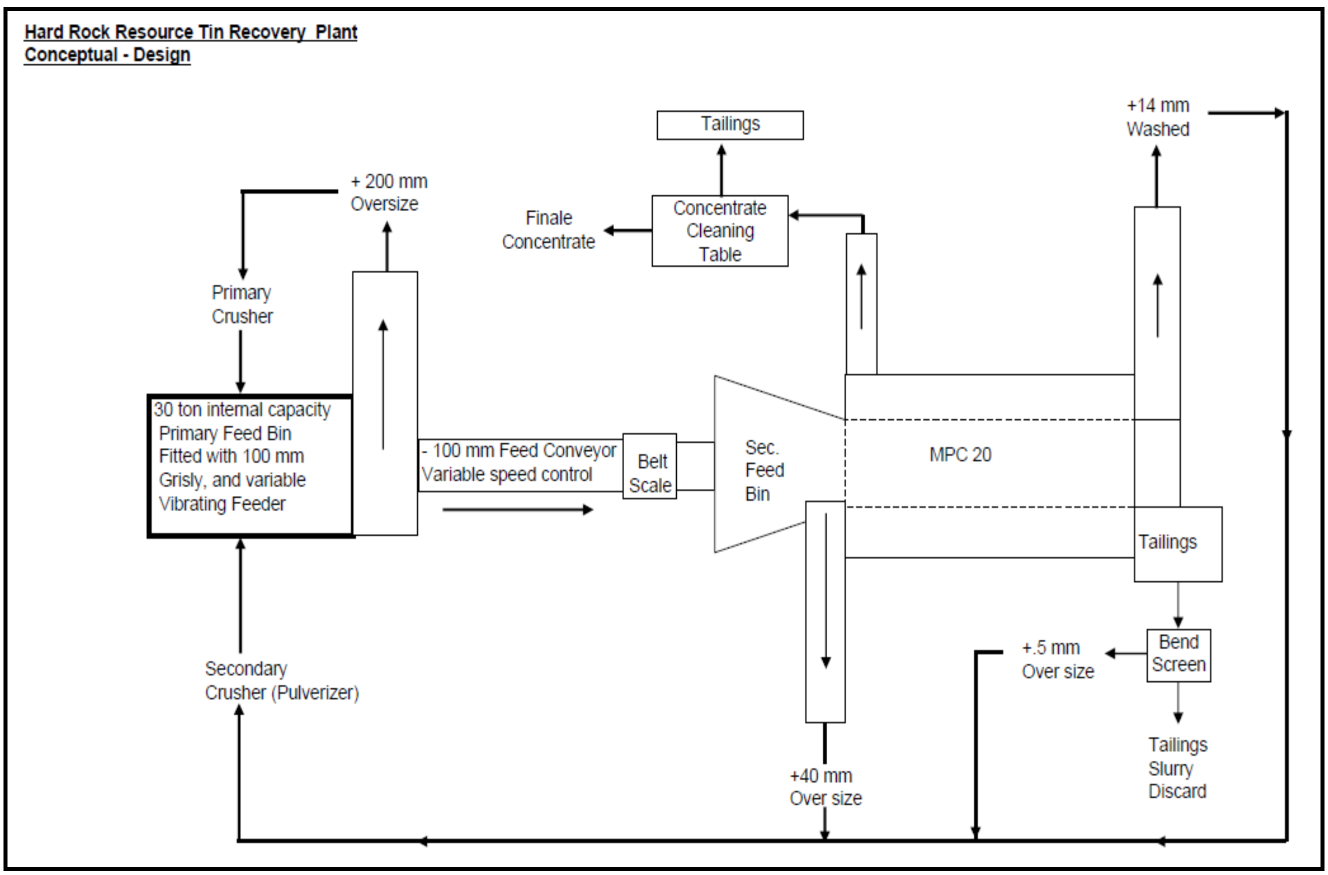

Fig 5.2: Conceptual design for a hard rock tin recovery plant, (after Valchem, November 2008)

Table 5.1: Vaalchem Proposed Budget for hard rock tin recovery plant

Hard Rock Resource - tin Recovery

Proposed Budget

Item	Cost (ZAR)
Primary feed bin	65,000
Oversize conveyor	26,000
Feed Conveyor	34,000
Belt Scale	40,000
MPC 20	310,000
Concentrate Conveyor	17,000
Concentrate cleaning Table	38,000
Table Tailings pump	24,000
+40 mm Oversize Conveyor	24,000
+14 mm Oversize Conveyor	24,000
Tailings sump pump	28,000
Bend screen	31,000
Tailings pump	38,000
+0.5 mm Oversize Conveyor	24,000
Primary Crusher	88,000
Secondary Crusher (Pulverizer)	84,000
Total	895,000

5.3 Conclusion

The range of gravity-concentrating equipment used for the recovery of cassiterite is quite extensive. Beneficiating processes using gravity should not be easily dismissed in favour of more sophisticated new technology which may be more-expensive processes. Mintek did not provide cost estimates for their tin extraction procedure. The plant design and cost estimates from Vaalchem were used in discounted cash flow (DCF) analysis in the next chapter.

CHAPTER 6: FINANCIAL MODELLING, RISK ANALYSIS AND MANAGEMENT

6.0 Introduction

Risk identification and management relates to the reduction of uncertainty associated with the resource and its conversion to reserve and a comprehensive system of control would be required to ensure that risks are identified, measured, assessed and mitigated or managed. In this report, a matrix of risks is created which reports on the main areas of risk, tolerance levels and mitigation strategies for the Tin Project. Financial modelling is used as one of the means for quantifying, assessing and managing risks associated with the Tin Project. An explanation on how risks, tolerance levels and mitigation strategies change as the project progresses through its various stages of development from its present dormant stage to an operating mine is given.

6.1 Risk classification

Mining project risks are classified into systematic (financial) and unsystematic (technical) risks. The risks are explained in detail in the following sections.

6.1.1 Systematic or financial risks

The systematic risks are essentially financial risks and are associated with financial and capital markets. They relate to market uncertainty and its effect on cost of capital. They also relate to risk of financial failure of the project because capital and interest (cost of capital) cannot be repaid from the cash-flow. The risk may also arise from the financier and include external influences such as interest rate hikes, market collapse, inflation and currency exchange (Fig 6.1.). The management has no direct control of these risks, however, they can be accounted for through discount rate adjustments. Thus financial risks are accounted for in financing arrangements and factored into cost of capital (e.g. Weighted Average Cost Capital). The risks should be known and factored into financial analysis (Macfarlane, 2004).

6.1.2 Unsystematic risks or technical risks

Unsystematic risks are technical risks, which are mainly project and operational risks and are directly related to inputs of business and include orebody uncertainty and technological uncertainty. As in the case with systematic risk, unsystematic risks may arise from external influences such as interest rate hikes, market collapse, inflation and currency exchange (Fig 6.1.).

The internal unsystematic risks can be dealt with individually and at source and can be mitigated by diversification to minimum levels (Macfarlane, 2004).

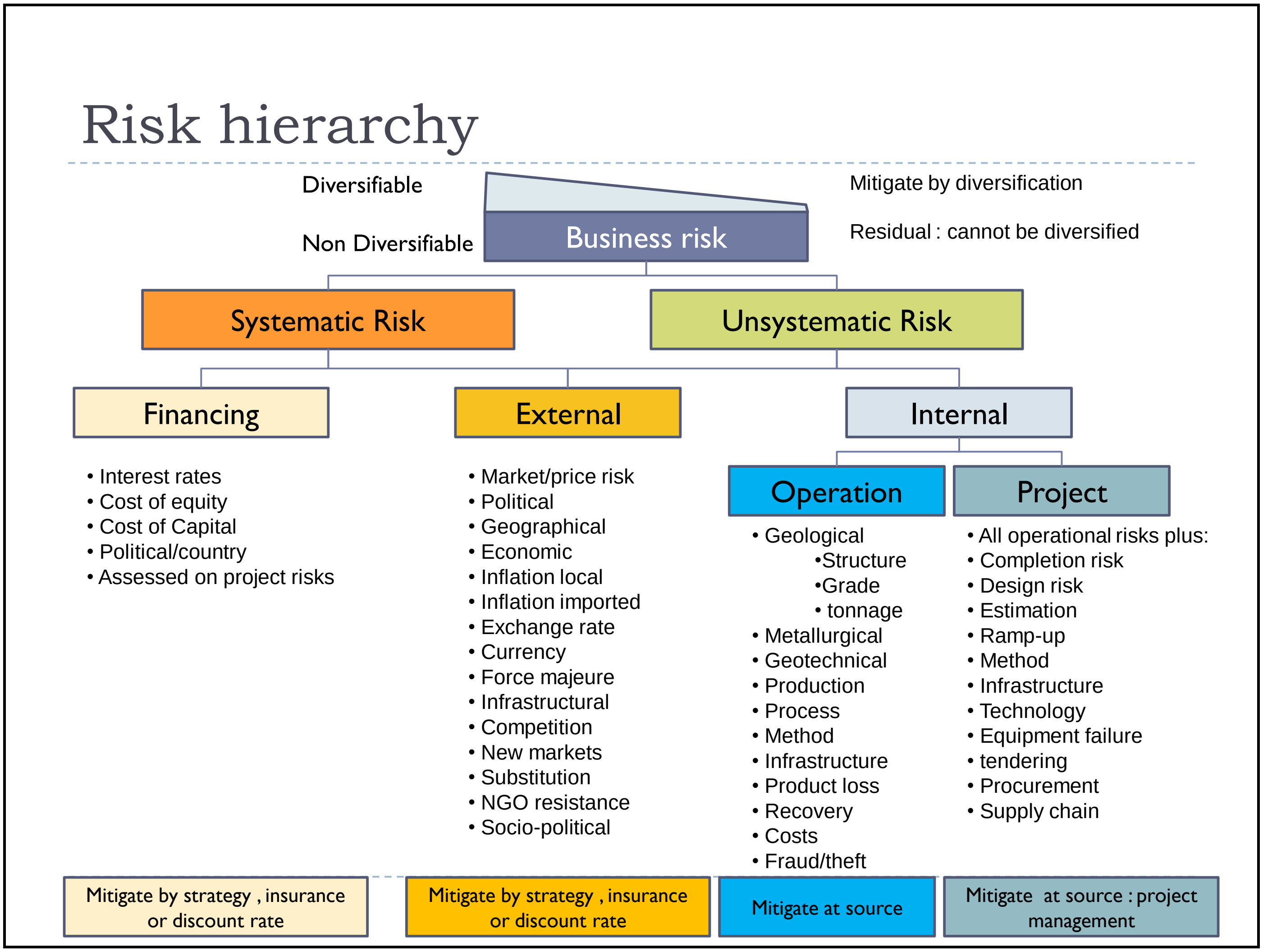

Fig 6.1: Risk classification, (after Macfarlane, 2009, slide presentation)

6.2 Risk quantification

Risk is defined as the product of its probability of occurring and its impact. Impact can be quantified quite simply by estimating its effect on cash flow when it occurs to its full extent, in a "what-if-situation", (Macfarlane, 2004).

6.3 Risk assessment

Risk assessment can either be qualitative or quantitative. A detailed description of the risk assessment approaches is given in the following sections.

6.3.1 Qualitative risk assessment

Qualitative risk assessment identifies the risk and assigns subjective ranges for impact and probability. The method is inaccurate but it allows complete identification of risks and prioritisation of risk management focus areas to be done. It is useful to adopt a systematic qualitative approach for project reviews where residual risk areas can be quickly identified based on the experience of the reviewer or technical experts (Macfarlane, 2009).

6.3.2 Quantitative risk assessment

Quantitative risk assessment includes:

- Matrix Approach
- Sensitivity analysis
- Best case/Worst case Scenario
- Simulation
- Decision Tree Analysis
- Real Option Analysis or Modern Asset Pricing Technique

Best case/worst case Scenario, Simulation, Decision Tree Analysis and Real Option Analysis are probabilistic approaches to risk as they require quantification of probability of occurrence in mathematical terms (Macfarlane, 2004).

6.4 Main areas of risks (risk identification)

The identification of risk is done by knowledgeable experts in the field. In the case of the Tin Project, both systematic and unsystematic risks were identified as listed in Fig 6.1, above. The tin project is still at pre-feasibility stage especially considering the amount of historical information available. Appendix 4, shows main areas of risks, quantification and possible mitigation strategies.

Apart from the risks in Appendix 4, additional financial and technical risks have been noted as shown in Fig 6.1. Financial risks such as the cost of capital, interest rate hikes, etc have been noted. Technical risks such as geological uncertainty, e.g. grade, tonnage and structural risks have been identified. In addition, metallurgical, infrastructure risks, etc have been noted as shown in Fig 6.1.

6.5 Risk tolerance levels or uncertainty criterion

The risk tolerance levels for the financial variables and estimates should be in the order of those listed in Table 6.1. The Tin Project's tolerance levels should be in the form of those highlighted in green since the project can be considered as being at either exploration level or scoping study since the existing data has not yet been validated by additional drilling. However, when the existing data is honoured then tolerance levels in the prefeasibility study are acceptable at the level of indicated resource, Table 6.1. The confidence in the financial estimates increase with increasing project life and therefore increasing knowledge accumulation as shown in Table 6.1 and the trend is explained in section 6.6 as shown in Fig 6.6.

Table 6.1: Acceptable risk levels at each stage of project study, (after Macfarlane, 2004). MARR – Minimum Accepted Rate of Return

Level of study	Resource Category (upper limit)	Capital Estimates	Contingences	Cost Estimates	Revenues	Inflation	Discount Rate	Confidence Levels
Exploration	inferred resource	>30%	30%	>30%	>30%	No	N/A	50-75%
Scoping	inferred resource	20-30%	25%	20-30%	20-30%	Yes/No	MARR	75-80%
Prefeasibility	Indicated resource	10-20%	15%	10-20%	10-20%	Yes	MARR	80-85%
Feasibility	Probable reserve	5-10%	10%	5-10%	5-10%	Yes	MARR	85-95%
Bankable Feasibility	Probable reserve	5-8%	8%	5-8%	5-8%	Yes	WACC	90-95%
Production Schedules	Proven reserve	<5%	5%	<5%	<5%	Yes	WACC	95%

The confidence levels shown in Table 6.1 are basically the Project's tolerance levels. At exploration stage, if there is at least 50% chance of success, the project will progress forward. Conversely, if the chance of success is less that 50%, the project would be abandoned unless there are measures to improve the chance of success.

6.6 Risk mitigation strategies

Although the tin project has considerable amount of historical data, it is still at scoping study since permits have to be issued before first stage of infill drilling and sampling commences. Risks have been identified qualitatively and assessed qualitatively and listed in a risk registry. A risk matrix has been tabulated with probability of occurrences and impact to establish a risk rating. Both probability and impact are scaled out of 5 and the risk rating is the product of the two. A rating of 25 would be such that the risk is too high to be tolerated. A risk rating of 0 would imply that the impact is negligible. Appendix 4 is a matrix of identified risks and their risk ratings when there are both no measures to mitigate them and when measures are put in place and the probability of risk of occurrence reduced. An example would be the risk of not getting exploration permit. There is a more or less 50:50 per cent chance of either getting the permits or not getting the permit. However, the impact is very high (qualitatively) and is assigned a value of 5. The 3/5 chance of not getting the permit is reduced by meeting all the Department of Minerals and Energy (DME) application requirements on application and making all the necessary follow-ups. The same logic approach is used for all the other risky areas (Appendix 4).

6.6.1 Risk matrix

The risks that are most relevant to the Tin Project are listed, quantified and mitigation strategy has been put in place as shown in Appendix 4. The technical risks will be mitigated at source once the permits have been issued. This will include further in-fill drilling and sampling, modelling including (geostatistical modeling) to ascertain the dimension, grade and tonnage of the mineral resource. More work will be done to mitigate metallurgical risk through pilot plant investigations. A programme should be put in place to ensure that value is added at each stage throughout the value chain. However, additional work should be done at each stage of the value chain to gather more information which will be used to make more informed decisions once permits have been issued. The systematic or financial risks would be mitigated by strategy and through discount rates.

6.6.2 Financial modelling and sensitivity analysis

The substantial information on this project makes it possible to perform sensitivity analysis in order to identify the impact of the most variable parameters on this project. A standard DCF and sensitivity analysis was carried out in order to identify crucial parameters to be considered in decision making process. The inputs to the DCF were obtained from existing similar sized mining operations to establish the capital and operating costs. Tin recovery was based on the plant design by Vaalchem which is also comparable with that established from previous mining operation at Zaaiplaats Tin Mine although this may improve with technological advancements. Cost estimates are listed in Table 6.2 (VMI, 2008) and it should be noted that these cost estimates may be wrong and may need to be reviewed.

Table 6.2: Capital schedules in nominal terms

Cost Parameter (ZAR)	Year 1	Year 2	Year 3	Year 4	Year 5
Prospecting permit	10,000				
Deskop studies and geological interpretation	25,000				
Development of geological model of existing data	25,000				
Reconaissance loam and stream sampling	50,000				
Geophysics and Geological Mapping	150,000				
Diamond drilling		250,000			
In-fill drilling			500,000		
Borehole survey		50,000	100,000		
Wire line logging		50,000	100,000		
Logging, Sampling, Assaying		150,000	300,000		
Analytical and metallurgical testwork		500,000	3,600,000		
Initial Interpretation and evaluation		50,000	50,000		
Environmental and community cost		167,000	283,000		
Pre-feasibility				500,000	
Geological modelling				250,000	
Final feasibility study and final report				250,000	
Tin Recovery Plant x 4 @ ZAR895 000 each rate 50t/hr					3,580,000
Sub-Total (ZAR)	**260,000**	**1,217,000**	**4,933,000**	**1,000,000**	**3,580,000**
Total (ZAR)	**10,990,000**				

The NPV and IRR for this project at average grade of 813 ppm Sn is shown in Fig 6.2. The project has an internal rate of return of 78.85%, an NPV of R 23,849,019 at a discount of 12%. As a first pass decision tool, this is a robust project and investment is encouraged to move forward. As an additional tool, the payback period indicates that the project will be able to pay back all costs in less than half a year from the start of production, which is 3.84 years from start of desktop studies, see Fig 6.3. The sensitivity analysis was performed on NPV using revenue, price, recovery, exchange rate, operating expenditure (OPEX) and capital expenditure (CAPEX). It was noted that a 10% change in the first 4 parameters (revenue variables) would reduce or increase NPV by 11.56 Million rands where as a 10% change in OPEX will change NPV by 7.68 Million and a 10% change in CAPEX will change NPV by 996 thousand rands. Thus NPV is less sensitive to Capex but considerably sensitive to price, revenue, recovery, exchange rate and operating costs. Thus mitigation will be required to reduce changes in these parameters through:

- hedging for price, exchange rates and revenue stability,
- trusted technology for plant efficiency and efficient management, efficient and effective management systems and strategy, tested efficient technology to reduce operational costs

The other financial risks have been mitigated through a nominal discount rate of 12%. The discount rate has been chosen on the assumption that this project is at pre-feasibility stage, and that financing costs and options have not been considered yet and thus, the discount rate contains a component of the project Minimum Acceptable Rate of Return (MARR), inflation and country/project risk (Macfarlane, 2004). In some cases it would be encouraged to take country risk insurance but this would not seem necessary for South Africa with a relatively low country risk rating.

NPV	IRR
R 46,501,959	
R 41,409,034	
R 36,951,769	
R 33,038,139	
R 29,591,192	
R 26,546,310	
R 23,849,019	78.85%
R 21,453,222	
R 19,319,766	
R 17,415,278	
R 15,711,217	
R 14,183,095	
R 12,809,833	
R 11,573,230	
R 10,457,530	
R 9,449,048	
R 8,535,870	

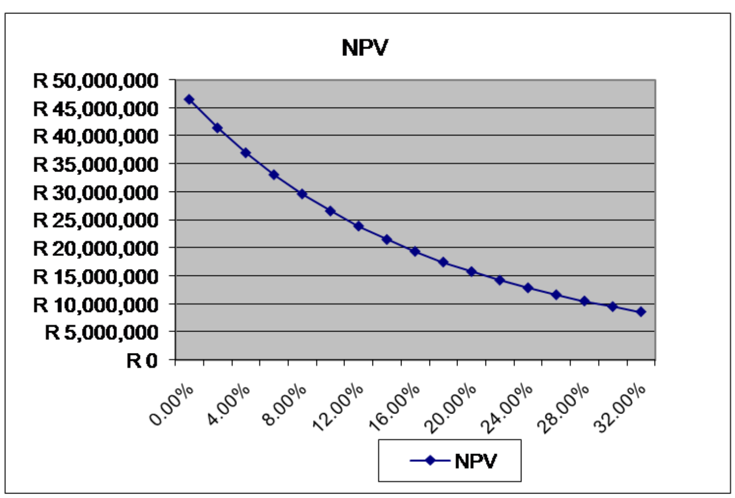

Fig 6.2: NPV and IRR derived from polygonal average grade estimate of 813ppm Sn

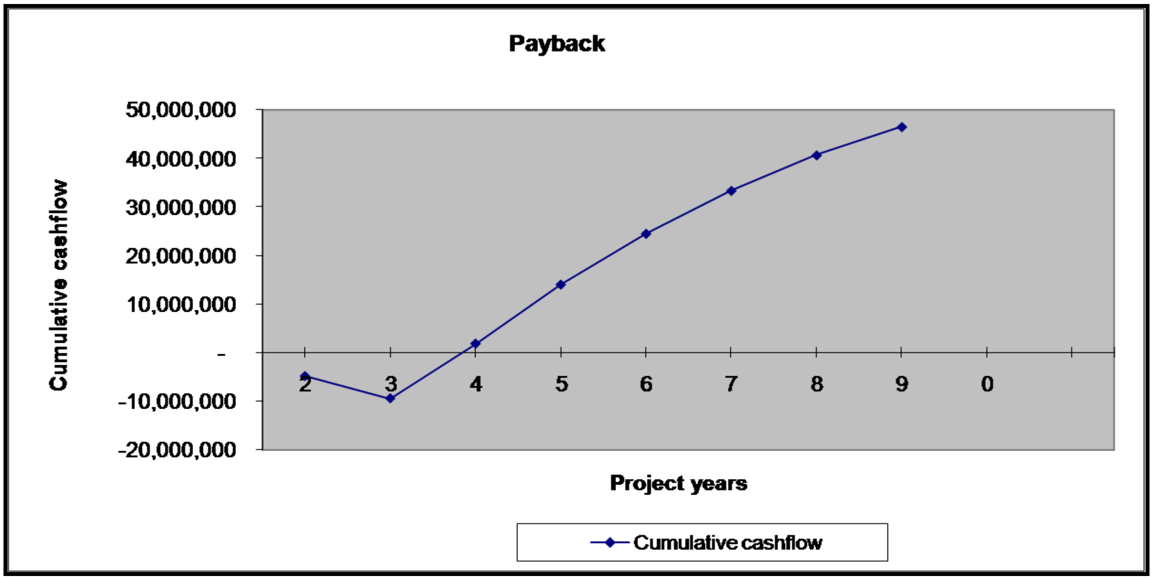

Fig 6.3: Payback period for the Tin Project at 813 ppm average grade (3.84 years).

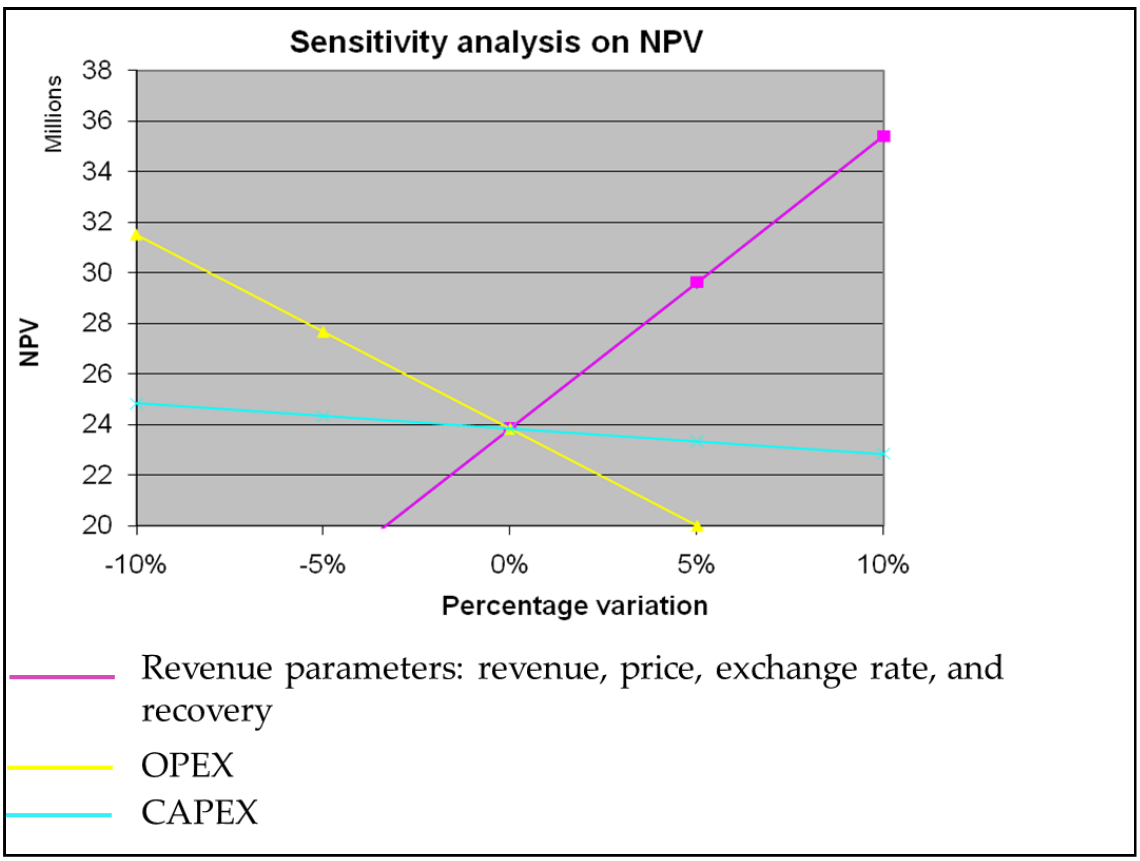

Fig 6.4: Sensitivity analysis for the Tin Project

6.6.3 Scenario analysis

Net present value (NPV) and Internal rate of return (IRR) were calculated for both polygonal average grade estimate of 813ppm Sn and kriged block average grade estimate of 528 ppm Sn.

The average grade of 813 ppm gives a positive NPV of 23.8 million rands and an IRR of 78.85% at a discount rate of 12% (Fig 6.2). These results are highly encouraging and investment in this project is recommended to go forward.

On the other hand, the NPV derived from the kriged block average grade of 528 ppm Sn is negative, i.e. (-21.0 million rands), which is highly discouraging.

In order to achieve the average grade of 813ppm, selective mining methods would be recommended. Bulk mining is discouraged because it has the effect of homogenising and lowering the grade and renders the project uneconomic as shown by the NPV derived from the kriged block estimate of 528 ppm Sn (Fig 6.5).

NPV	IRR
R -35,483,701	
R -32,178,980	
R -29,311,934	
R -26,814,065	
R -24,628,892	
R -22,709,644	
R -21,017,429	#NUM!
R -19,519,776	
R -18,189,475	
R -17,003,633	
R -15,942,925	
R -14,990,976	
R -14,133,864	
R -13,359,714	
R -12,658,367	
R -12,021,098	
R -11,440,396	

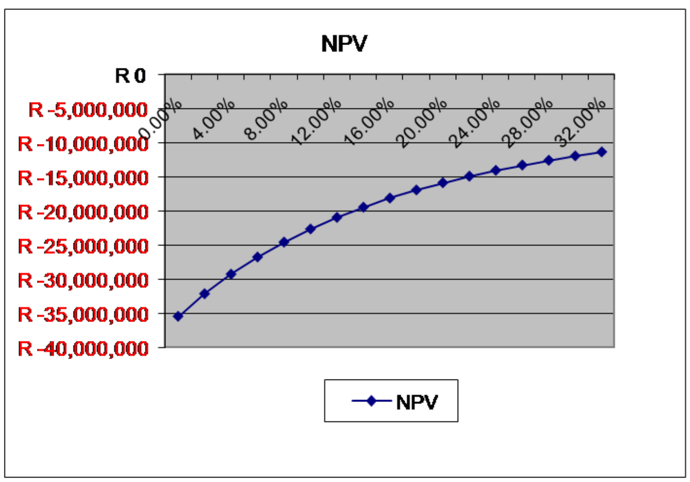

Fig 6.5: NPV and IRR derived from kriged block average grade estimate of 528 ppm Sn

6.6.4 Recommendations

It is recommended that, further studies should be done that will analyse risk and capture management flexibility and the study should include the following:

- Mont Carlo Simulation DCF analysis which will help quantify the probability of occurrence of the risk areas identified by sensitivity analysis. Simulation DCF will analyse risk, however, it does not capture management flexibility.
- Decision Tree Analysis will reflect management flexibility.
- Real option Valuation will capture value. It will assess and reflect price flexibility. (Macfarlane, 2009).

6.6.5 Financial and technical risk profiles

The financial and technical risks, discussed in the preceding section follow a pattern as shown in Fig 6.6 below. Level of risk for financial risk will decrease during the life of a mining venture whereas that of technical risks will increase with life of the mining project. Technical Risks in mining projects increase with life of project because of resource depletion, grade becomes lower and lower, tonnage becomes less and less. Operational and processing efficiency decreases because the machinery becomes older and older. Financial risks decrease because the estimation of costs, prices, inflation, etc become more and more certain, more accurate and become known with high confidence.

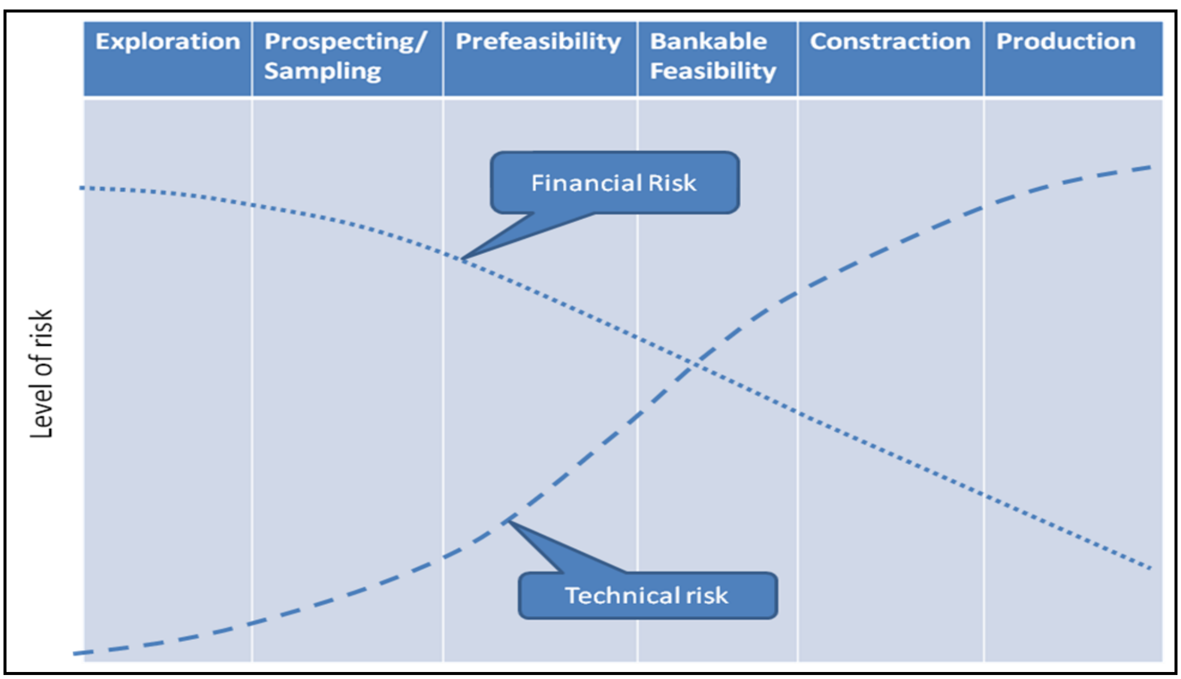

Fig 6.6: Financial versus technical risk during life of a mining project (after Kornelius and Forbes,2003)

6.7 Market considerations

The tin concentrate, which can be in the order of 55 % tin can be sold FOT (Free-On-Trucks), ex plant and transported by purchasers. At the moment (December, 2009), there is no mine in South which is producing tin, thus the Tin Project , if successful, will supply more than 80% of Tin demand in South Africa. In the past, the export market has not been exploited due to local demand. Prior to its closure in 1978, Zaaiplaats Tin Mine was the sole producer of Tin in South and 80% of its production was consumed by South African's largest metal company, ISCOR and the rest by other local steel manufacturers (Zelda Real Estates, 2006). There is growing demand for the metal in China and India in the iron and steel industry to support the current industrialisation in these countries and this explains the increasing price of tin after the recession that started in March 2008. The current price of tin is US$15,000/ton (October 2009) which may rise due to growing demand and this makes the Tin Project a potentially viable investment proposition.

6.8 Conclusion

The tin project should have a matrix of risks that are relevant to the stage of the project with the main areas of risk, tolerance levels and mitigation strategies; an example is presented in Appendix 4. The Tin Project has a reliable historical dataset, which although not yet been tested and validated, has been used to show that the project is robust from the DCF's NPV and IRR when an average grade of 813ppm Sn is considered in the case of selective mining. Sensitivity analysis has identified main areas of risks which will require attention and mitigation in future. An explanation for how technical and financial risks change as the project progresses through its various stages of development from its present dormant stage to an operating mine has been given. DCF on its own shows expected future cash flow but does not assess risk and management flexibility. However, sensitivity analysis and scenario DCF have helped to analyse risk areas that need attention, although the methods do not show management flexibility. Simulation DCF, Decision Tree Analysis and real Option Analysis are other capital budgeting techniques that can be used to analyse risks and to ascertain the feasibility of the Tin Project. In addition, Decision Tree Analysis can capture management flexibility.

CHAPTER 7: DISCUSSIONS AND CONCLUSIONS

7.1 Discussion

The tin resource on Roodepoort-Groenfontein prospect has been modelled technically and financially using computer assisted modelling techniques. MapInfo was used to model the geology and for geostatistical evaluation of the resource in question. The evaluation of the Tin Project has considered a resource which is covered by overburden of less than or equal to 30m, although a few blocks of less that 33m were incorporated in the block estimation. A cut-off grade of 200 ppm Sn was chosen based on the grade continuity.

The total tonnage for the deposit has been estimated to be about 6 Million tons using both polygonal and geostatistical estimation. The polygonal grade estimate is 813 ppm Sn whereas the kriged grade estimate is 528 ppm Sn. The difference in grade estimates between the two methods is mainly attributed to the fact that the polygonal estimation incorporates two high grade samples (RDP080 and 146), which had a strong effect of raising the overall average grade. The kriged estimate removes the effect of these two high grade samples because they were not used in computing the variogram model that was used for kriging. It is generally common practice to mask few high grade samples in creating the variogram models but then include the high grade samples when kriging.

Financial modelling using discounted cash flow (DCF) analysis has shown that the resource could be economically viable if selective mining approach is used to attain an average grade of 813 ppm Sn obtained from polygonal estimation. The project would not be economically viable at 528 ppm Sn as estimated by block kriging and this would suggest that bulk mining method would render the project uneconomic. Sensitivity analysis has indicated that NPV is highly sensitive to changes in revenue parameters (exchange rates, tin recovery and price) and operational costs. Attention will need to be focused on these variables in order to reduce risks associated with these parameters. An example would be to avoid stripping cost of mining 1 million tons of ore covered by 10 to 30m overburden and use an alternative mining method. The remaining 5 million tons of mineralisation is covered by less than 10m overburden which can be removed at low cost and thus improve the NPV of the project.

Tin recovery can be improved by use of efficient and trusted technology as well as efficient and effective workforce. If both technical and financial risks are identified, assessed and mitigated, the Tin Project can be an economically viable investment proposition.

7.2 Conclusions and recommendations

The analysis of 1978 tin assay results from Roodepoort-Groenfontein mineral property has shown potential for the existence of an economic disseminated tin mineralisation. Considering a cut-off grade of 200ppm and a maximum overburden thickness of 30m, anomalous disseminated tin-bearing body within the Lease Granite has an indicated resource of 6.26 million tons of mineralisation averaging 813 ppm Sn (polygonal estimate) and the resource would yield 5,085.81tons of tin. The kriged estimate has shown existence of an anomalous disseminated tin-bearing ore body which has an indicated resource of 6.06 million tons of mineralisation averaging 528 ppm Sn and the resource would yield 3,203.86 tons of tin. The project is economically viable at 813 ppm Sn average grade with a positive NPV of 23.8 Million rands and an IRR of 78.85%. At average grade of 528 ppm Sn, the project is not economically viable with a negative NPV (-21.02 Million rands). The project NPV can be improved through risk identification, assessment and mitigation to reduce costs and add value.

Additional value adding, risk assessment approaches such as best case/worst case scenario, simulation, decision tree analysis and real option analysis can be used since these are probabilistic approaches to risk as they require quantification of probability of occurrence in mathematical terms. Management flexibility would also add value for example by ensuring that efficient selective mining is properly conducted with grade control measures in place. Value adding expenditure such as conducting further studies e.g. in-fill drilling and sampling to validate the drilling of 1978 and to fully investigate full thickness of the resource is recommended. Mineralogical test-work and pilot plant investigation is recommended to ascertain the plant design, recovery and size, which is important in establishing a life of mine plan. A proposed low cost gravity self separation is underway and should the new technology be implemented it would enhance the economic viability of the tin resource. Value can be added by analysing possible by-products such as fluorspar and tungsten.

The technical and financial evaluation of the tin resource using computer assisted methods has been a successful exercise. The results obtained suggest that the tin resource could be economically viable and further investigations to ascertain cost and capital parameters is recommended at feasibility stage.

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