A technical risk evaluation of the Kantienpan volcanic hosted massive sulphide (VHMS) deposit and its financial viability

by

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DECLARATION

I declare that the thesis that I hereby submit for the Masters Degree in Earth Science Practise and Management at the University of Pretoria has not previously been submitted by me for degree purposes at any other university.

SIGNATURE OF STUDENT: ________________ DATE: 2003/07/01

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I give thanks to my Saviour, Jesus Christ, for giving me the ability, commitment and perseverance to complete this project.

Job 1: 1-12

"Surely there is a mine for silver, and a place for gold to be refined. [2] Iron is taken out of the earth, and copper is smelted from ore. [3] Miners put an end to darkness, and search out to the farthest bound the ore in gloom and deep darkness. [4] They open shafts in a valley away from human habitation; they are forgotten by travellers, they sway suspended, remote from people. [5] As for the earth, out of it comes bread; but underneath it is turned up as by fire. [6] Its stones are the place of sapphires, and its dust contains gold. [7] "That path no bird of prey knows, and the falcon's eye has not seen it. [8] The proud wild animals have not trodden it; the lion has not passed over it. [9] "They put their hand to the flinty rock, and overturn mountains by the roots. [10] They cut out channels in the rocks, and their eyes see every precious thing. [11] The sources of the rivers they probe; hidden things they bring to light. [12] "But where shall wisdom be found? And where is the place of understanding?"

Job 28: 28

"And he said to humankind, 'Truly, the fear of the Lord, that is wisdom; and to depart from evil is understanding.' "

(The Holy Bible : New Revised Standard Version. 1989. Nashville: Thomas Nelson Publishers.)

ABSTRACT

The Areachap Group represents a mid-Proterozoic fossil island arc environment consisting of amphibolite, hornblende gneiss, quartz-feldspathic gneiss, calc-silicates and pelitic schists. Chemical compositions of these highly deformed upper amphibolite/granulite grade metamorphosed rocks indicate protoliths ranging from rhyolite/rhyodacite, calc-alkaline basalt, tholeiite to ultramafic igneous rocks and sediments. The above-mentioned assemblage is typical of an island arc environment.

Island arc environments are ideal hosts for volcanic hosted massive sulphide (VHMS) type deposits and may successfully be explored by using the VHMS lithogeochemical alteration model. VHMS deposits not only yield strategic base metals such as zinc (Zn), copper (Cu) and lead (Pb), but significant grades of gold (Au) and silver (Ag) are associated with these deposits.

The Areachap Group presents a metallogenic province containing one economic deposit, the Prieska Zn-Cu mine, as well as several sub-economic deposits, including the Areachap mine and other lesser prospects at Boksputs, Kantienpan, Jacomynspan and Rokoptel. The Prieska Zn-Cu mine is the most significant VHMS deposit of the Areachap Group and occurs within the Copperton volcanic centre. This abandoned mine delivered 47 Mt sulphide ore at 1,7 % Cu and 3,8 % Zn with traces of Ag and Au.

Four volcanic centres were previously identified in the Areachap Group, namely Upington, Klein Begin, Boksputs and Copperton. Exploration activities were loosely subdivided into the same regions. Regional lithogeochemical sampling campaigns were conducted for the four subproject areas and approximately 5 000 rock samples were analysed for the twelve major oxides and ten trace elements.

The region of interest, the Boksputs Subvolcanic area, with a well-established infrastructure, is situated near Groblershoop (50 km east) and Marydale (30 km

southeast) in the Northern Cape province and is part of the geological Areachap Group.

Several high copper anomalies and the tholeiitic lithological composition of the Boksputs Subproject resulted in this area being selected as the main target region. It was attempted to discriminate between different trace element populations using probability plots, but this was not successful. The complexity of the probability plots was attributed to the large variation in different rock types included in the data set. Corrections were made by determining threshold values for each rock type, but this refinement proved unsuccessful, indicating that the rock classification used was incorrect. Option areas were finally selected, based primarily on absolute Cu values. These areas were mapped in more detail prior to ground electromagnetic (EM) surveys and drilling. To test the target selection, a proto-lithological map of the area, based on cluster analyses of the lithogeochemical dataset, was drawn. The proto-lithological maps formed the basis of the follow-up work and the application of the VHMS conceptual model.

A conductor in the Kantienpan target area was located with a time domain electromagnetic (TDEM) survey and this was drilled. The drilling intersected a massive sulphide body with a tonnage of approximately 5 Mt and an average grade of 4.09 % Zn, 0.49 % Cu and traces of Au and Ag.

The orebody was evaluated financially and it was found to be uneconomic as a stand-alone operation. However, if the Kantienpan deposit is considered as an alternative to imported concentrate for the Zincor smelter, this study suggests that the project may be economically feasible. Furthermore, it must be stated that the Areachap Group remains only partly explored and that a world class VHMS deposit may be discovered within the next few years.

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1 INTRODUCTION

1.1 GENERAL

During 1994, Zn was highlighted as a strategic commodity for Iscor. The company's management set a target to find zinc deposits to sustain the current zinc operation, Rosh Pinah, and possibly enlarge Iscor's portfolio. Most of the global zinc produced at this stage was from volcanic hosted massive sulphide (VHMS) deposits and the obvious objective was to find a blind VHMS deposit with economic grades that would be suitable for an underground mining operation.

The use of lithogeochemical exploration to locate blind VHMS orebodies was developed and applied successfully in especially Canada and Australia (Urabe, Scott and Hattori, 1983; Whitford and Ashley, 1992). There appears to have been relatively little effort to locate VHMS deposits by this means in South Africa. Previous exploration efforts by Iscor Ltd., were aimed at the more viable, bigger-tonnage sedimentary exhalative (Sedex) deposits. A full understanding of both the VHMS and Sedex conceptual models in the mid-1970s came at a time of peak activity in exploration, especially in the Proterozoic Northern Cape terrain. A waning interest in sustained base-metal exploration in South Africa followed this period. In addition, electro magnetic (EM) technology has advanced considerably since the mid-1970s (Cain, 1994).

Falconbridge initiated exploration programmes, utilising this technology, in the greenstone belts of Botswana (Tati and Matsitama) from 1978 to 1981 and in Zimbabwe (Bulawayo, Gatooma-QueQue-Hartley and Bindura-Shamva) from 1980 to 1982. The primary target was gold, rather than VHMS deposits. Funds were in extremely short supply with Falconbridge, especially for VHMS exploration, and budget cutbacks were frequent. There were thus no long-term initiatives to apply and test the Canadian approach on a grass-roots basis (Cain, 1994).

The Maranda Zn-Cu deposits in the Murchison Belt (Terblanche, 1997), Limpopo Province, some Anglo American prospects in the Barberton greenstone belt of Mpumalanga, the closed Prieska Zn-Cu mine of the Northern Cape, Cactus Cu-Pb-Zn mine in Zimbabwe, Bushman in Botswana and Elbe, Matchless and Otohahasi in Namibia appear to be the only VHMS deposits in Southern Africa. This apparent paucity of VHMS deposits is in strong contrast to the high incidence of such deposits in favourable geological environments. This suggests that, since VHMS deposits are the most important source of zinc (Cain, 1994), a more active exploration effort for these deposits is warranted.

The entire Proterozoic Areachap Group in the Northern Cape, which represents a fossil island-arc environment (Geringer *et al.*, 1994), was targeted by Iscor. An intensive three-year exploration programme for blind VHMS deposits commenced at the beginning of 1995, which was extended by another year and a half, owing to personnel shortages.

Lithogeochemical sampling techniques were used for initial blind target identification, and constituted Phase 1 of the exploration work. Phase 2 and 3 work comprised of TDEM and detail TDEM surveys, drilling target definitions and evaluation in the third and fourth years, respectively (Rossouw, 1999).

Four possible geochemically and structurally related, calc-alkaline stratovolcanic centres make up the Areachap belt, viz. Upington, Klein Begin, Boksputs and Copperton centres (Middleton, 1976).

The Areachap belt's importance lies in the numerous VHMS base-metal sulphide showings hosted within its 250 km-outcrop length. Included is the more recently closed-down 47 Mt (3,87 % Zn, 1,74 % Cu [+8,0 g/t Ag and 0,4 g/t Au]) Prieska Zn-Cu mine of Anglo Vaal. The other more significant prospects were summarised by Humphreys (1985) (Appendix A) and include Areachap, Jannelsepan, Klipbakke, Boksputs, Kantienpan, Van Wyks Pan, Edenville, Kielder and Hedley Plains (Voet and King, 1986 and Theart, 1985).

1.2 STUDY OBJECTIVES

The objectives of this study are to document various aspects of the geology of the Kantienpan deposit and environs, to provide additional tools for exploration of VHMS deposits and to determine the economic viability of the Kantienpan deposit.

1.3 LOCATION

The project area is situated in the flat, arid Northern Karoo region of the Northern Cape Province of South Africa (Figure 1). The initial project area was a block 250 km by 40 km, covering the Areachap Group from Areachap mine, north of Upington, to Prieska Zn-Cu mine, southwest of Prieska. The area was then narrowed down to the few farms under option, shown in Figure 2.

The farms under option cover an area of approximately 40 km by 20 km. This area lies approximately 75 km south-south-east of Upington, 45 km west-south-west of Groblershoop and 50 km east-north-east of Kenhardt (Figure 2).

1.3.1 Location Risk

As far as the location of the project is concerned, no particular risks exist, except that it is far from any smelters.

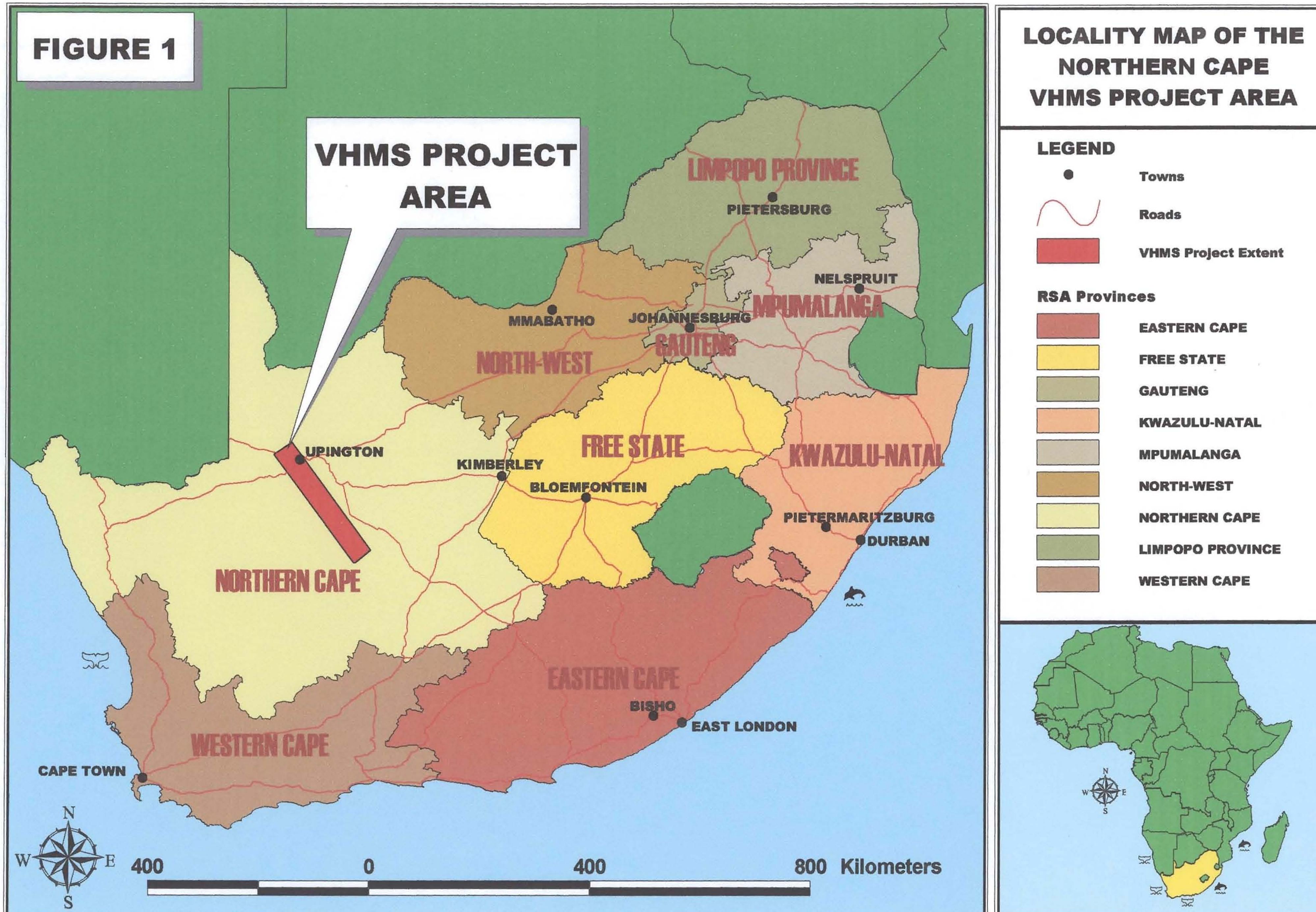

Figure 1. Locality map of the Northern Cape VHMS project area.

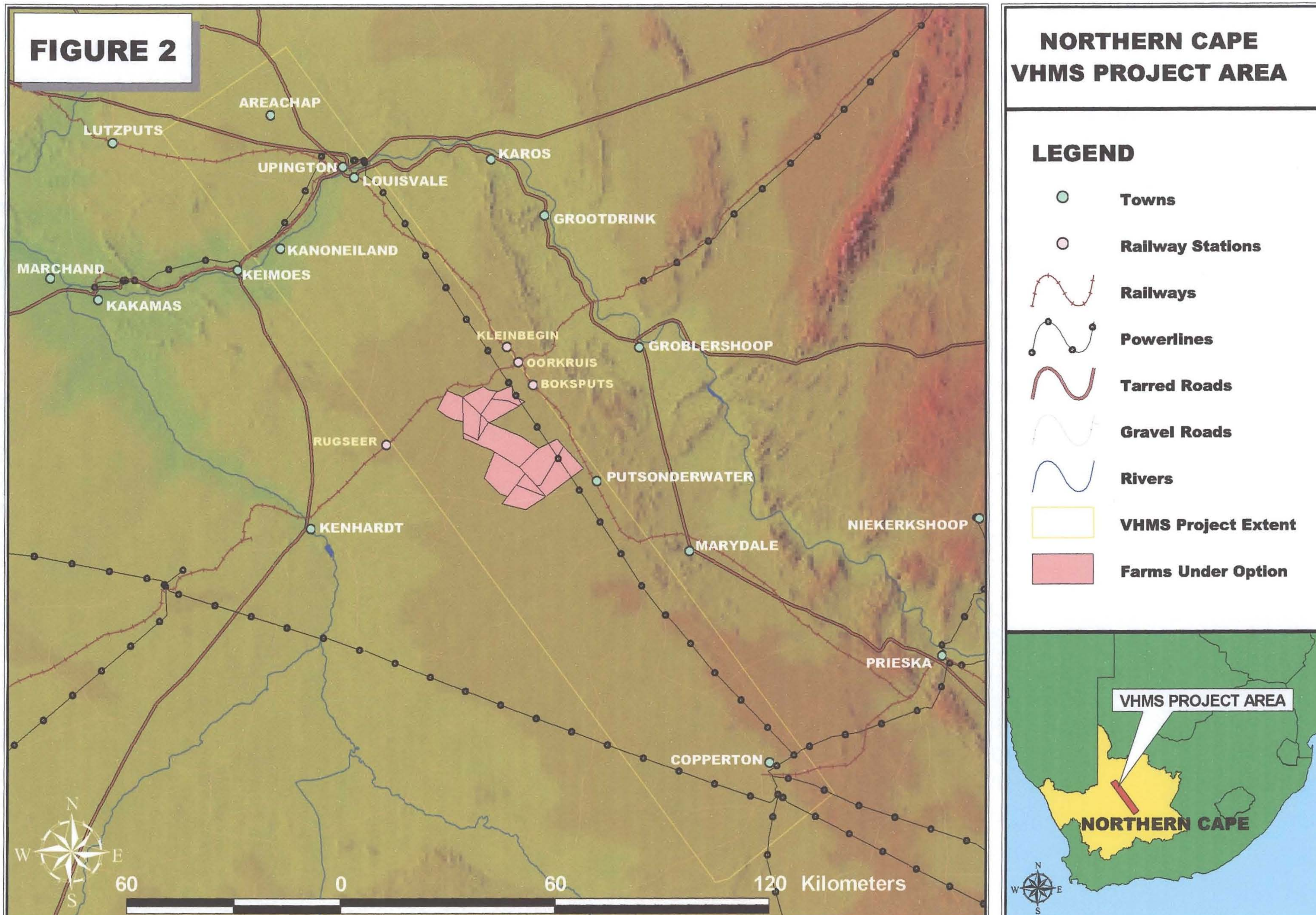

Figure 2. Locality map and local infrastructure of the VHMS project area.

1.4 INFRASTRUCTURE

The project area has a well-established infrastructure. Two major tarred roads and a network of good gravel roads traverse the area (Figure 2). Farm tracks vary from moderately good to very bad and some barely exist. Kenhardt and Groblershoop are respectively 53 km and 56 km from the project area via a well-maintained, gravel road (Rossouw, 1999).

The Upington-De Aar railway line runs along the eastern border of the Areachap Group project area, with the Boksputs station being the closest railway link at a distance of 18 km. The east-west Sishen-Saldanha railway line cuts across the belt directly north of the project area, with Kleinbegin station being the closest connection, also at 18 km (22 km via gravel road) (Figure 3). These two stations are equipped with loading facilities and, being on different railway lines, diversify the shipping option (Rossouw, 1999).

A power line also runs parallel to the Areachap Group project area, lying immediately to the east, with the closest point to the line being 8 km and the distance to the nearest substation being 13 km (Figure 3).

Telephone lines and microwave towers (telecommunications), as well as radio transmission towers and masts and cellular phone network masts, provide a good communication network in the area.

A well-equipped airport is situated at Upington with daily flights to Cape Town and Johannesburg via Kimberley. Two small airports also exist at Kenhardt and Prieska, but are only used for light aircraft. Public and private landing strips are also present in the vicinity of the project area and at the small settlements along the Orange River.

The area is arid and sparsely populated. Very few homesteads, of which some are often unoccupied, occur in the region. The nearby towns, however, host a variety of tourist and accommodation facilities like game and hunting lodges, hotels, caravan parks, guest houses, 4x4 trails, motorbike trails, hiking trails and golf courses.

1.4.1 Infrastructure Risk

Water supply for a concentrating plant may become a problem. The base case scenario is that sufficient groundwater can be extracted from drill holes and the mining operations to supply the plant's demand. The worst-case scenario is that a 40 km water pipeline must be built from the Orange River, should there not be sufficient water to supply the plant.

1.5 PHYSIOGRAPHY

Local topography is relatively flat within the project area. To the east of the Kheis Sub-Province there is a prominent range of mainly quartzite hills. To the west, the topography of the northern Karoo and Bushmanland is typically flat.

Farming activities include extensive sheep and limited beef cattle husbandry, with some dairy, poultry and ostrich farming taking place. Along the Orange River, at Upington and to the east of the project area, irrigation allows the large-scale cultivation of cash crops such as watermelon and melons, as well as the development of wheat, cotton and lucerne lands, date plantations, and particularly sultana, wine and table grape vineyards.

Altitudes of between 800 m and 1 100 m are recorded. Small amphibolite and amphibole gneiss hillocks rise up to about 50 m above the plains.

The mean annual rainfall of 200 mm occurs mainly as heavy thunderstorms in spring and late summer. Extremely hot summers and cold winters with frost are experienced.

Vegetation is sparse. Typical vegetation consists of open scrubby savannah with occasional acacia (e.g. Camel Thorn) and *Boscia albitrunca* (e.g. Witgat) where water is available. Thick scrubby vegetation is present in low-lying alluvium-filled drainage beds.

1.5.1 Physiographical and Environmental Risk

The rehabilitation and self-regeneration potential of the very sensitive desert type vegetation is low and, therefore, disturbance should be restricted to the absolute minimum. Vegetation clearing should be avoided and, where this is unavoidable, measures must be implemented to avoid loss of topsoil through uncontrolled run-off. Should erosion start, it must be stopped by installing gabions or other methods to break the velocity of the water and dissipate rivulets into smaller streams.

An environmental management programme report and fire management plans must be completed and approved by the relevant provincial conservation authority before any development may proceed.

1.6 LAND TENURE

The relevant magisterial district is Kenhardt, which falls under the Gordonia Regional Services Council.

A summary of the mineral and surface right owners is given in Table 1 and can also be seen in Figure 3. All the portions of land mentioned in Table 1 were explored with the appropriate exploration permit. The title deeds of the different properties are not appended to this document.

Table 1. Mineral and surface right holders of the VHMS project area.

Farm Description	Mineral & Surface Rights Owners	Extent
Boksputs 118, Portion 7	A Strauss	3646.4051 ha
Boksputs 118, Portion 8	WP Stauss	1892.0203 ha
Boksputs 118, Portion 9	WP Stauss, T Strauss, M Niehaus	3284.7894 ha
Zonder Pan 173, Remaining Extent Zonder Pan 173, Portion 5	F Lacock	4286.1992 ha 4282.8694 ha
Kantienpan 119, Remainder of Portion 1	F Kruger	4282.6612 ha
Kantienpan 119, Remaining Extent Kantienpan 119, Portion 2	Schmidt Family Trust, S Schmidt	9396.6493 ha 409.7029 ha
Gemsbok Bult 120, Portion 2	Broer Visser Family Trust	4329.9477 ha
Rooi Puts 172, Portion 2 Rooi Puts 172, Portion 3	D Malan	1131.1804 ha 3747.2690 ha
Rooi Puts 172, Portion 4	K Visser	6929.1981 ha
	TOTAL : 47 618.892 ha	

1.6.1 Tenure Risk

In terms of the new minerals legislation (Mineral and Petroleum Resources Development Act, 2002) all mineral rights will be owned by government, which implies that the Kantienpan mineral option could no longer be exercised in future. Should the decision be taken to mine the deposit at Kantienpan, a mining permit or extended exploration permit must be obtained within a year after the act has come into power, otherwise the mineral right may be lost to another company. It is recommended that applications should be filed for the necessary permits.

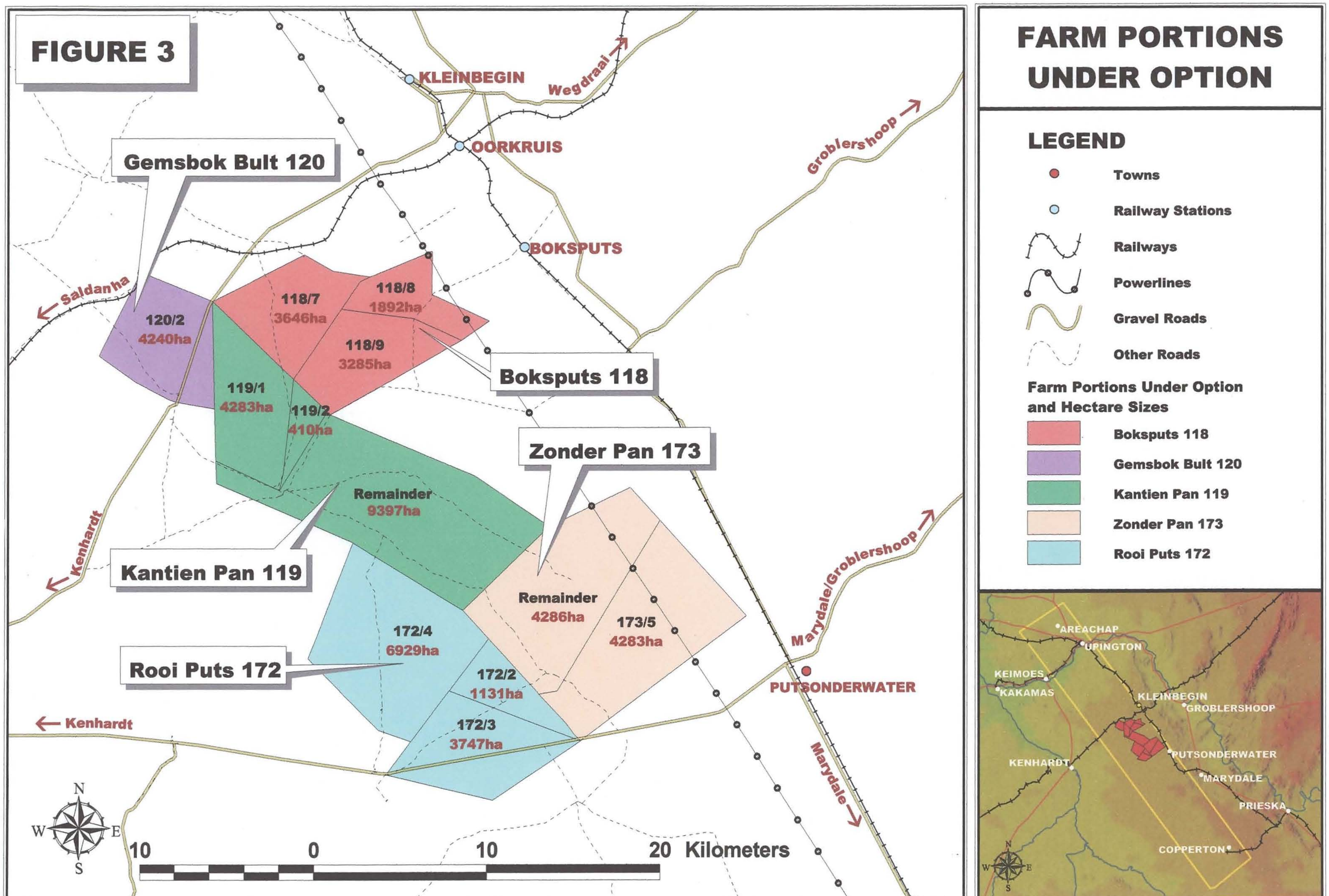

Figure 3. A map of farms taken under option within the project area.

2 HISTORY

2.1 PREVIOUS WORK ON THE ECONOMIC GEOLOGICAL ASPECT OF THE REGION (1969 – 1994)

2.1.1 Introduction

Iscor Ltd, lately Kumba Resources Ltd, assigned Cain to undertake an extensive desktop and literature survey into the economic potential of the Areachap Group (Cain, 1994). He requested the help of Blignault and Van Schalkwyk for Zn and Cu target generation within the Areachap Group and their results are summarised in Appendix A (Blignault and Van Schalkwyk, 1995). These surveys indicated that numerous companies, including Anglo American (AAC), Phelps Dodge, Anglovaal, JCI and a few others have explored the region. The central part of the belt, however, seemed to be very poorly known or understood from a potential economic point of view (Blignault and Van Schalkwyk, 1995).

The last active base-metal prospecting ventures took place mainly in the early/mid-1970s with some work done in the early 1980s. Most of this early exploration was done at a time when the VHMS conceptual model had not attained full maturity. Some authors, however, started using the model, or parts of it, as we know it at this stage in time, to explain their deposits. Middleton, for instance, applied the model to the Prieska Zn-Cu mine in 1976, Gorton used this model to explain Kielder, in his thesis in 1981, and Voet and King used the model at the Areachap deposit in 1986.

Lithogeochemical sampling, as an exploration tool and of this magnitude, has probably never been applied in previous exploration campaigns, since this type of sampling only started in the early 1990s.

Mineralised targets of the past, for example Kielder (Gorton, 1981), were all of a surface or sub-outcropping nature and defined by regional soil/gossan geochemical surveys and follow-up geophysical (magnetic, IP, pulse EM and gravity) surveys and relatively shallow drilling (Gorton, 1981 and Gresse, 1978). Conceptual volcanic-strata mapping was limited, to virtually non-existent, except on a deposit scale. In addition, the then available EM geophysical methods of application had very limited penetration capability. Regional airborne electromagnetic (INPUT and McPhar 400) and magnetic surveys sometimes missed the bodies owing to parallel and not perpendicular flight lines to the strike of the geology. Some deposits were however found successfully, as in the case of Jacomynspan and Prieska Annex (Attridge, 1986; Blignault and Van Schalkwyk, 1995).

2.1.2 Known Deposits

Known deposits and base metal occurrences are briefly described below and localities are shown in Figure 4. The deposits are listed and described in greater detail in Appendix A.

2.1.2.1 Upington area

Exploration was previously undertaken by Iscor on the Areachap deposit (Figure 2) with the objective to mine massive pyrite portions of the Areachap body for sulphuric acid production. The Areachap copper-zinc massive sulphide is one of two similar VHMS deposits found in the belt; the other being Anglo Vaal's 47 Mt Prieska Zn-Cu mine.

The Areachap deposit was discovered in approximately 1885 due to the recognition of mineralisation in gossan float (Rogers and Du Toit, 1908). Drilling of Areachap from the 1960s indicated resources of 1,6 Mt massive sulphides to between 200 m and 300 m depth (including a reserve of 0,5 Mt grading 2,3% Zn, 0,5% Cu and 42% sulphur, considered to be a mining prospect). Shaft-sinking to 200 m was carried out, with stations cut at 91, 122, 152 and 182 m levels and a cross-cut at 91 m (which intersected the old 1909 – 1917 workings for supergene-enriched sulphide). The project was deferred with the discovery of the Prieska Cu-Zn deposit (Voet and King, 1986).

Cape Asbestos obtained exploration rights from Iscor in 1971 and, in a joint venture with AAC, proved 8,1 Mt grading 2,4% Zn and 0,54% Cu to 750 rn depth by diamond drilling between 1971 and 1973 (Voet and King, 1986).

Cape Asbestos ceded its rights to AAC in 1974, which in turn relinquished these back to Iscor in 1977 after further geological and geophysical work and drilling had failed to locate additional mineralisation. AAC was of the opinion, however, that it would be possible to locate extensions to the present body by deep drilling (Voet and King, 1986).

The Jannelsepan sulphide body of presumed VHMS affinity occurs crudely on strike with Areachap on the south side of the Orange River (Geringer, 1994).

2.1.2.2 Klein Begin and Boksputs areas

A number of small base metal mineral occurrences have been reported for this area, which include Klipbakke, Boksputs, Kantienpan, Van Wyks Pan and Edenville (Theart, 1985).

It is known that Shell undertook exploration work in the Boksputs centre until the early 1980s, but no details are known.

2.1.2.3 Copperton area

The Prieska Zn-Cu mine commenced mining operations on the massive sulphide at Copperton in 1972 with reserves of 47 Mt grading at 1.74% Cu and 3.87% Zn, 8.0 g/t Ag and 0.4 g/t Au. The massive sulphide is 1 to 30 m thick, extends along strike by 2 000 m and persists to a depth of 1 000 m. The massive sulphide consists of various lenses and is strata-bound within fine-grained, laminated gneisses of the Copperton Formation. Sulphide minerals include pyrite, sphalerite, pyrrhotite and galena. Traces of Ag and Au are present (Middleton, 1976; Theart, 1985 and Wagener and Van Schalkwyk, 1986). The mine was closed during 1991.

A copper lode with a north-westerly strike is shown on the Rogers and Du Toit regional geological map of 1910, but the economic potential was only realised in 1968 (Theart, 1985). After its discovery, Middleton did the first comprehensive description of the Prieska Zn-Cu mine, in a thesis in 1976.

The Annex Cu-Zn massive sulphide deposit was discovered in 1969 and is situated 5 km south of the Prieska Zn-Cu mine. Chlorite-biotite schist correlated with the Copperton Formation hosts the deposit. The ore reserves are estimated to be 1,5 Mt of 1,5 % Cu and <0,5 % Zn (Middleton, 1976; Wagener and Van Schalkwyk, 1986 and Blignault and Van Schalkwyk, 1995).

Other occurrences in the Copperton centre include Kielder and Hedley Plains. Newmont carried out regional prospecting in the Kielder area in the late 1970s, making virgin discoveries (Gordon, 1981 and Gresse, 1978).

The massive Zn, Cu and Pb sulphides reported from portions of Kielder included one lens delineated (K3) and two only partly delineated (K1 and K6) by diamond drilling. Using a 2,5% Zn + Cu cut-off for the K3 body, an estimate of 1,3 Mt grading 4.32 % Zn, 0.33 % Cu and 4.45 g/t Ag was arrived at. A crude estimate of K6 gave 1 Mt grading 8.85 % Zn, 0.42 % Cu and 15 g/t. The highest Pb value in some of the core was in the order of ±1 %. The Newmont report of 1978 states that three anomalies (K2, K4 and K5) remained to be investigated (Gorton, 1981 and Gresse, 1978). K2 and K4 had only been drilled with one diamond hole each and disseminated pyrite mineralisation was found.

The K7 gossan was disclosed by soil geochemistry, but not drilled. Widespread gossan float was noted on the southeastern portion of Grassmoor, bordering Vogelstruisbult, as well as on Eureka (Gorton, 1981 and Gresse, 1978).

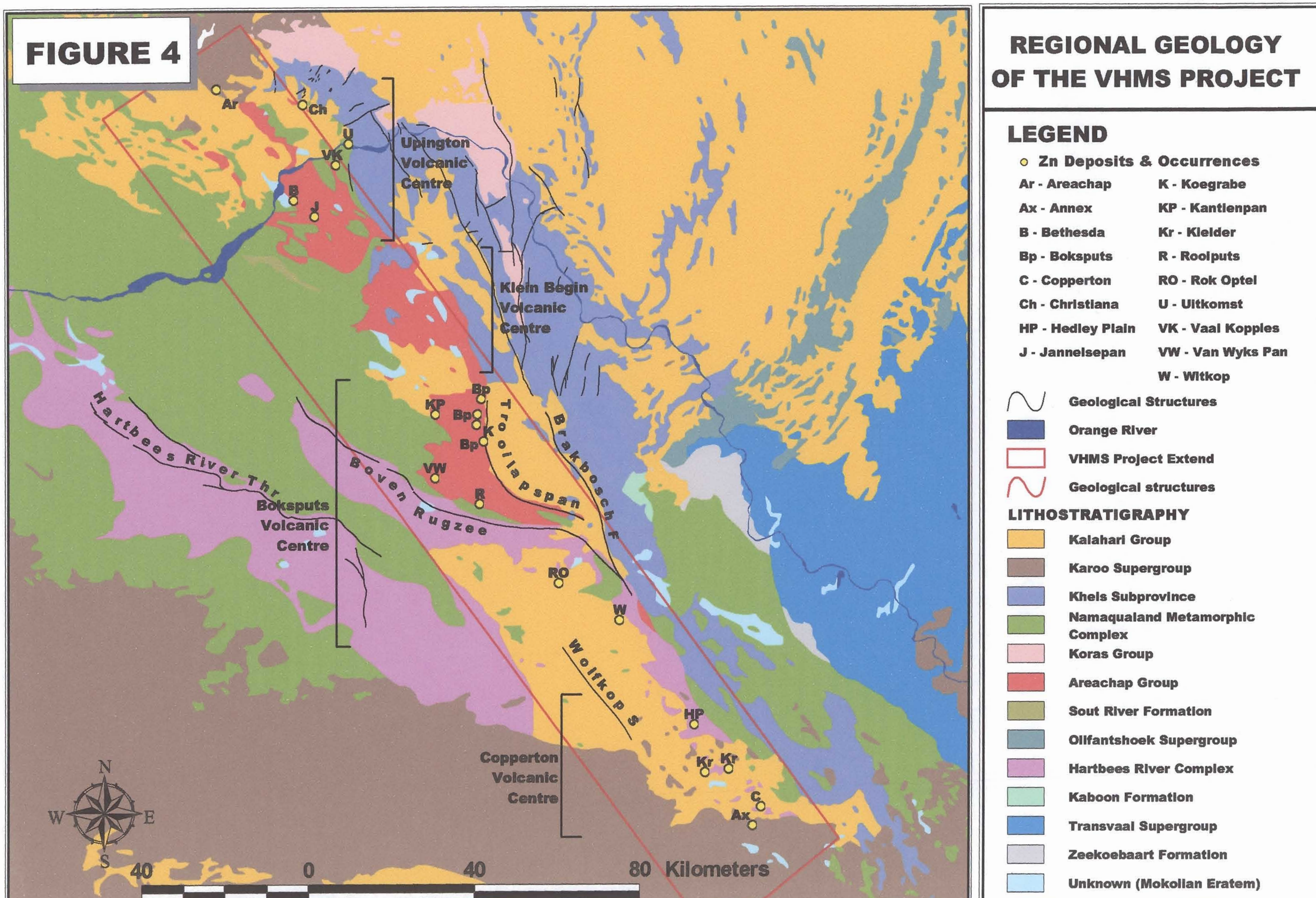

Figure 4. Regional geological map of the VHMS project (Modified after Geringer et al., 1994)

2.2 WORK DONE FOR THIS STUDY

2.2.1 Introduction

The Areachap Group volcano-sedimentary stratigraphic package was targeted for VHMS deposits based on Cain's application of VHMS models as used by Falconbridge and updated by Australian researchers (Cain, 1994).

Four possible volcanic centres were proposed by Middleton (1976) and Geringer (1994), and these are show in Figure 4. Regional lithogeochemical sampling was done of these centres comprising the entire Areachap Group (250 x 40 km), from Areachap Mine in the north to Copperton in the south (Figure 5). Regional geological maps (1:25 000) were then prepared, using the data gathered along the lithogeochemical sampling lines and photo geological interpretation of aeromagnetic survey data supplemented by published geological maps and literature. Lithogeochemical maps were prepared which incorporated the alteration zones and which also fitted the exploration model's criteria (Appendix C). Protolithological maps were prepared by means of lithogeochemical parameters and geostatistical methods, which included cluster analysis (Rossouw and Geraghty, 1997).

The volcanic centres were prioritised by means of identification of fractionation trends i.e. tholeiitic vs. calc-alkaline, as well as Zn and Cu anomalies. The Boksputs centre was selected as first priority in consultation with Dr A Galley of the Canadian Geological Survey (Rossouw and Geraghty, 1997). A geographical information system (GIS) was used in generating targets within the Boksputs centre by applying the exploration model, which used alteration signatures, protolithology and geochemistry (Rossouw and Geraghty, 1997). Negotiations started in 1998 to obtain options for mineral rights on the prioritised farms which properties can be seen in Figure 3.

Detailed follow-up mapping of the targets on the optioned farms were done and limited grab rock sampling was used for geochemical confirmation. Detailed ground magnetic and TDEM surveys were conducted to prioritise targets, which were then tested with diamond drilling for confirmation and resource estimation (Rossouw, 1999).

2.2.2 Lithogeochemical Survey

Regional exploration began in January 1995, and started with the taking of lithogeochemical rock samples along roads and fence lines at intervals of 100 m, with a line spacing of 1 to 2 km and a sample density of 8 samples per km^2. Rock sampling positions are shown in Figure 5.

A total of 6 010 lithological rock samples were taken across the Areachap Group, with 1 279 samples being taken in the Upington area, 1 734 samples in the Kleinbegin area and 2 997 samples in the Boksputs area (Figure 4). A further 160 rock samples were collected from the entire Areachap Group area to provide a reference set for further geochemical exploration, while 24 samples were taken from the Areachap core (Geraghty *et al*, 1996).

Threshold analyses using probability graphs were done and the relevant geochemical intervals were established to highlight possible alteration chemistry. Problems were however encountered due to the geochemical differences among the four centres (Figure 4).

Whole rock analyses (oxides and traces) were conducted on the rock samples and the results for each element and various ratios were plotted on geochemical maps for each region. The following aspects were considered during this investigation (Large, 2001):

- Enrichment in Ba is an indication of the exhalative in the direct vicinity of the massive sulphide lens of a VHMS deposit.
- Zn is depleted in zones of regional leaching related to VHMS deposit.
- Na_2O/SiO_2 ratio is lower in the leached zone relative to rock unaffected by hydrothermal alteration related to VHMS deposit.

Rock types were described along the sampling traverses and recorded for later reference.

2.2.3 Geophysical Surveys

2.2.3.1 Aerial Magnetic Survey

Two aerial magnetic surveys were done over the area.

- Newmont conducted a survey in 1970 that was flown perpendicularly to the strike of the Areachap Group (1 km line-spacing). Structural lineament analysis was done using the geology as an overlay to locate volcanic lentoid piles. This did not work owing to the structural complexity of the areas.
- The Council for Geoscience conducted a survey (available in electronic format), but the survey was flown parallel to the strike of the Areachap Group, in other words with north-south flight lines.

The combination of the two data sets would enhance the quality of the interpretation, but it was impossible, owing to computing limitations.

2.2.3.2 Ground Magnetic and TDEM Surveys

Detailed ground magnetic (Geotron G9) and TDEM (Geonics EM37) surveys were conducted across the targets and identified TDEM anomalies were followed-up with more detailed TDEM surveys to a defined drilling target. The drilling targets were tested with diamond drilling in conjunction with core logging and additional lithogeochemistry (Appendix E). Down-the-hole TDEM surveys were done in all the holes and drilling commenced to estimate reserve (Rossouw, 1999).

2.2.3.3 Radiometric and Gravity Test Surveys

Radiometric and gravity test surveys were also done on some of the proved conductors (Rossouw, 1999). An additional ground magnetic survey was carried out over the sand covered area to map the underlying stratigraphy. This survey identified interesting features that should still be followed up with more fieldwork (Geraghty *et al.*, 1996).

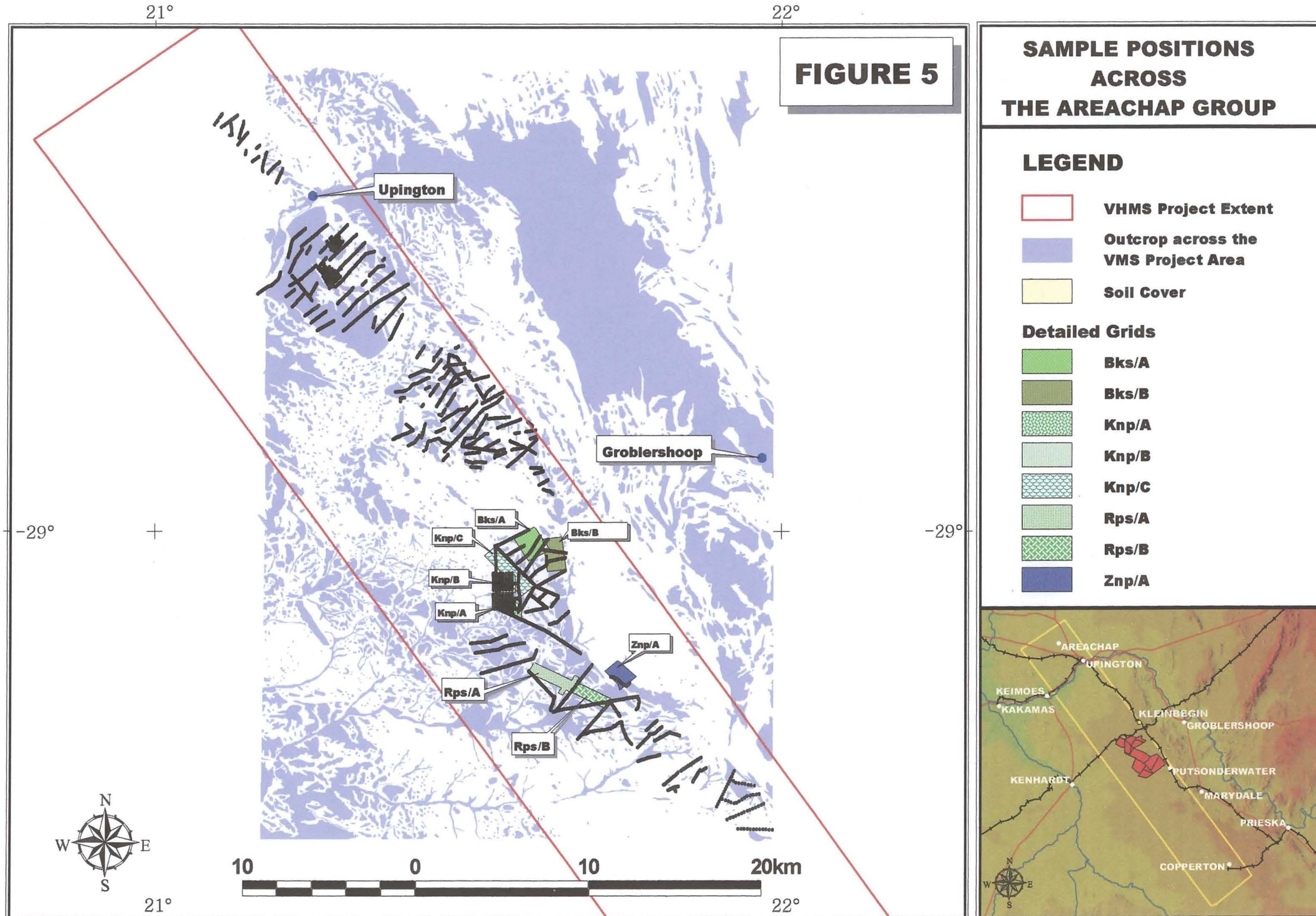

Figure 5. Rock sample positions across the Areachap Group.

2.2.4 Soil Survey

South of the Boven Rugseer Shearsone (Figure 4), the whole area is covered with sand and a decision was made by Iscor's management to conduct a soil survey across the area during 1995. Samples were collected every 100 m along the roads and fences on a 1 km line-spacing, which meant that approximately 1 000 samples were taken (Geraghty *et al*, 1996).

The purpose of the soil sampling exercise was to find gahnite, a dark-green to yellowish, zinc mineral of the spinel series. This mineral was found at the Prieska Zn-Cu mine and is believed to be a good indicator mineral for Zn mineralisation.

The following method was used:
Soil samples were collected every 100 m with a sampling size of 340 ml (cool drink can). Composites were made of every 10 samples collected over a distance of 1 km. The composite was then split with a riffler to finish with a 340 ml soil sample. Samples were washed and screened to dispose of the dust size fraction. Magnetic components were then removed with a magnet. Bromoform, with a density of 2.9 g/cm^3, was used to concentrate heavy minerals like pyroxene, amphibole, ilmenite and garnet in the dense fraction. This sample was then studied with a bi-ocular hand specimen microscope and it was found that 95 % of the heavy mineral fraction consists of garnet. Any blue or green minerals were handpicked and send for microprobe analysis. Unfortunately no gahnite was detected during this survey and it is suggested that the sampling technique employed, was not detailed enough to detect a mineral that may have a very limited dispersion in the secondary environment.

Other potentially negative aspects regarding this survey are the fact that the strike of the rocks was frequently not known and that it is not curtain if the sampling took place perpendicularly to the strike. Rock samples were also taken where outcrops occurred, but it could not be demonstrated that these samples contributed to a systematic interpretation of the region's lithology.

The heavy minerals programme was unsuccessful and abandoned (Geraghty *et al.*, 1996) and the conclusion was drawn that gahnite is so rare that it does not lend itself as a pathfinder mineral in this environment.

Further possibilities for this region are:

- Re-evaluate the samples to extract magnetite, pyroxene and amphibole for identifying enrichment in different areas that could be used as a mapping tool of specific lithological units.
- To find the possible existence of pyrrhotite and magnetite that is enriched in the massive sulphide zone, for which purpose a magnetic survey has already been conducted.
- Look for tourmaline, which is more abundant at Copperton rather than gahnite (Theart, 1985). Tourmaline can also be used as a pathfinder mineral and to map lithology.

A separate soil survey was conducted across parts of the lithogeochemical-surveyed areas. This reconnaissance mobile metal ion (MMI) soil survey was conducted to establish if this method is suitability for identifying base metal mineralisation (Appendix D). Traverses were sampled across known TDEM conductors to assist in prioritising these anomalies. The method proved to be successful and it is recommended that it should be considered in further exploration.

3 VHMS CONCEPTUAL MODEL AND EXPLORATION GUIDELINES

3.1 VHMS CONCEPTUAL MODEL

3.1.1 Introduction

There appears to have been relatively little exploration effort made to locate VHMS deposits in Southern Africa, with a few notable exceptions such as in the central zone of the Damara belt and the Barberton and Murchison greenstone belts. Probably almost no application of the lithogeochemical approach to exploration for these deposits has ever been undertaken. It is believed that much more exploration effort has been put into the more variable bigger-tonnage Sedex-model family of base-metal deposits. A full understanding of both the VHMS and Sedex conceptual models in the mid-1970s came at a time of peak activity in exploration. These exploration activities were especially concentrated in the Proterozoic northwestern Cape terrain, where after a waning interest in sustained base metal exploration in South Africa was seen. In addition, EM technology has since advanced considerably from the mid-1970s (Cain, 1994).

Falconbridge initiated lithogeochemical exploration programmes in the greenstone belts of Botswana (Tati and Matsitama) from 1978 to1981 and Zimbabwe (Bulawayo, Gatooma-QueQue-Hartley and Bindura-Shamva) from 1980 to 1982 with the primary focus on gold rather than VHMS deposits (Cain, 1994). Funds were in extremely short supply with Falconbridge, especially for VHMS exploration, and budget cutbacks were frequent. No long-term initiatives to apply and test the Canadian approach on a grass-roots basis were really possible.

The Murchison Cu-Zn deposits in the Limpopo Province, some AAC prospects in the Barberton greenstone belt of Mpumalanga, the closed Prieska Zn-Cu mine of the Northern Cape, the Cactus Cu-Pb-Zn mine in Zimbabwe and Elbe and Ochiase in Namibia, appear to be the only deposits in Southern Africa indicated as VHMS deposits. Their apparent paucity is in strong contrast to favourable geological environments. This suggests that a more pro-active exploration effort for VHMS deposits is warranted, because they are an important source of zinc (Cain, 1994).

3.1.2 Regional Setting

The Archaean Canadian and Palaeozoic (Caledonide) Norwegian VHMS deposits occur in submarine, predominantly mafic volcanic piles, e.g. greenstone belts and ophiolite-type successions (viz. Besshi and Cyprus subfamily).

Brunswick No 12 (Bathurst), British Columbia, Buchans (Newfoundland) and true Japanese Kuroko deposits also occur in submarine volcanic sequences, which comprise of bimodal mafic to felsic compositions, on continental crust; with marine sediments usually comprising more than 40% of the total succession. Mineralisation in VHMS deposits is always a result of sub aqueous exhalation, commonly by 'black smokers', of metal-rich hydrothermal solutions derived from fractionation of predominantly calc-alkaline magmas.

VHMS deposits occur throughout the geological record as:

- 'Primitive' Archaean greenstone-belt-hosted types (mainly the younger aged belts of circa 2,7 Ga, e.g. Abitibi belt of Canada);
- Proterozoic, Palaeozoic and Mesozoic types, less common; and
- Cainozoic, modern island-arc types, typified by true Kuroko-style deposits, very common in Circum-Pacific or Alpine tectonic terrain.

Primary features of VHMS deposits include (Figure 6):

- Massive ore 'lenses' (A zone) underlain by carrot-shaped, tapering 'stringer' ore zones of variable character (B2 zone);
- Characteristic depositional structures and textures; mineralogical and metal zoning;
- Alteration with constant geometric form is ubiquitously developed in immediate footwall of massive or in stringer ore as crosscutting pipe-like zones (B1 zone); and as
- Extensive stratigraphic conformable zones (C zone) on a huge (often tens of kilometres strike) scale parallel to the sea floor. These different stratigraphic layers can range from altered highly permeable layers (C zone) to the thin exhalite layer directly above the massive sulphide lens.

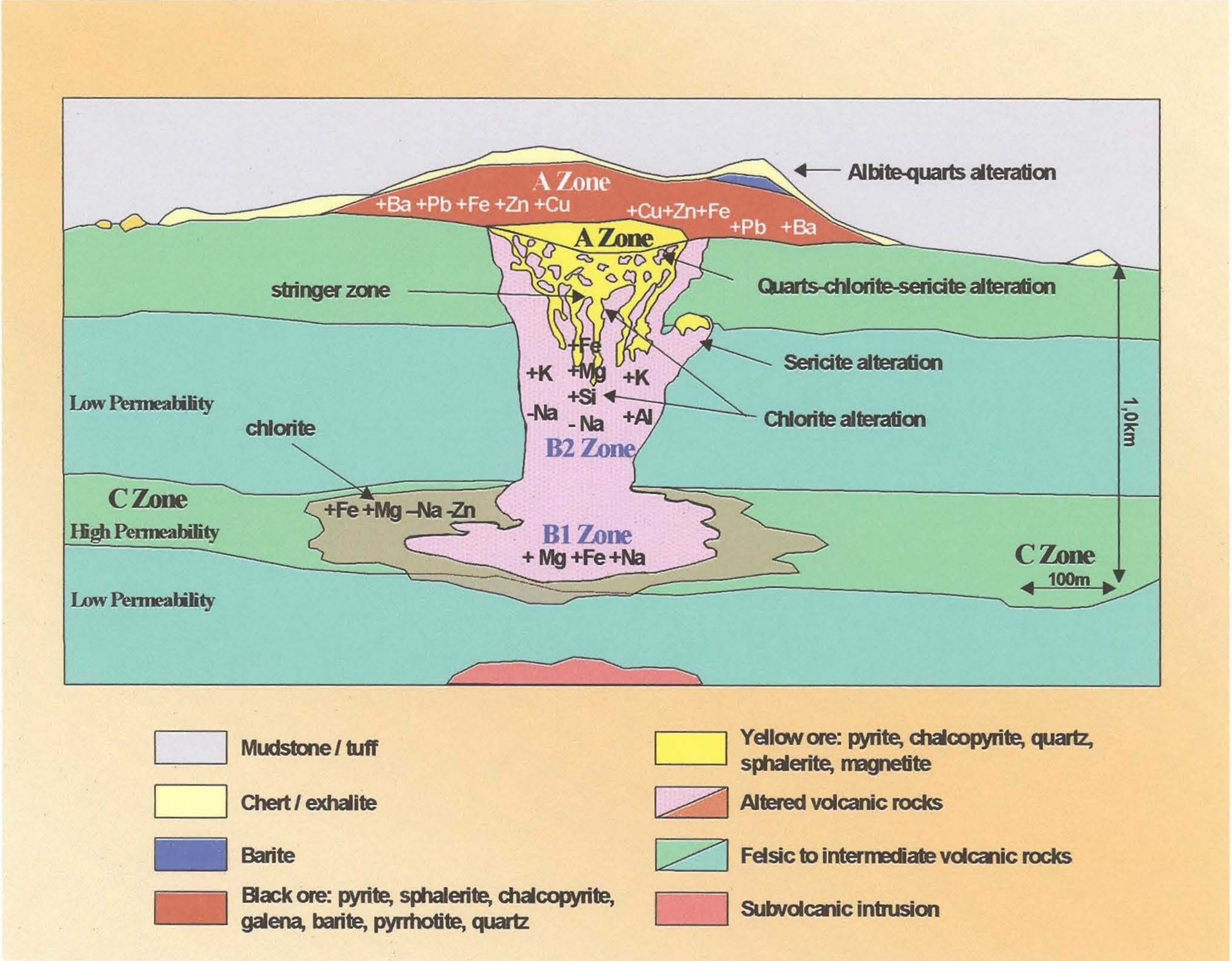

Figure 6. VHMS conceptual model (modified after Large et al, 2001)

These features may often be modified by subsequent metamorphic episodes and polyphase deformation, which impart secondary ore textures and affect metal distribution and shape (Franklin *et al.*, 1981 and Vokes, 1969).

3.1.3 Tonnage and Grade Considerations

VHMS deposits are an important source of the world's base metals, especially zinc (Appendix B). A short summary of typical average tonnage and grades of VHMS deposits are listed in Table 2.

Table 2. Average and median tonnage and grade of VHMS deposits (Cain, 1994).

Country	Region	Size Mt.	Zn %	Pb %	Cu %	Ag g/t	Au g/t
CANADIAN	Abitibi belt [n = 52]	9,2	3,43	0,07	1,47	31,7	0,81
	Bathurst camp [n = 29)	8,7	5,43	2,17	0,56	60,03	0,47
WORLD	(2) MEDIAN 50 percentile	1,5	2,0	0,0	1,3	13,0	0,16
	10 percentile [n = 432}	18,0	8,7	1,9	3,5	100,0	2,3

As in the case with MVT base-metal deposits, the VHMS family is not characterised by large tonnages on the whole. It does, however, like MVT deposits, also often occur in clusters. Unless a fairly systematic and meticulous approach is adopted with respect to its detection (and exploitation), there is always the possibility of smaller orebodies being 'missed', as clustering is common laterally – with deposits occurring in 20 to 40 km diameter range at a specific stratigraphic level/horizon. Vertical stacking of smaller deposits is also known; this being fairly common in Falconbridge's Millenbach mining district (Riverin and Hodgson, 1980 and Cain, 1994).

3.2 COMMON EXPLORATION GUIDELINES FOR VHMS DEPOSITS

The common, more important broader guides for locating VHMS deposits, mostly applied in conventional exploration, are discussed in the following paragraphs under the relevant subheadings.

3.2.1 Tectonic Controls

Lydon (1984) suggested that preferred geotectonic environments for VHMS deposits are near plate margins. Ophiolite associated deposits may reflect mid oceanic ridges or spreading back arc basins at divergent plate margins for example the deposits of Cyprus and Newfoundland. The Kuroko deposits of Japan and the Spanish-Portuguese pyrite belt, on the other hand, are associated with convergent plate margins in island arcs and continental margins. Intra-plate oceanic islands or more enigmatic plate tectonic environments also host deposits, such as those represented by Archean greenstone belts (Lydon, 1984). With the above-mentioned statements, is it essential to recognise geochemical associations related to extensional tectonism in continental areas.

A magmatic heat source is necessary below each complex to start the convecting seawater system. Deposits occur in clusters near subaqueous volcanic vents and/or 'black smokers'. Gravity can detect vertical tectonic domains in pristine environments. Transform offsets (and triple junctions) may control gross distribution of mineralisation.

At low metamorphic grade the ore bodies are flattened, with plunge:strike ratios often 3:1. At higher grades plunge:strike ratios increase to 10:1 (giving them a small surface area; therefore making them difficult to detect by mapping) (Lydon, 1988).

Detailed studies of metal zoning in deformed bodies, may assist in the understanding of style of mineralisation (e.g. Geco deposit in Canada – possibly relevant to the Prieska Zn-Cu mine, the Upington belt and Elbe in Namibia) (Stowe, 1983).

It is however important to note that the Areachap Group has undergone upper amphibolite metamorphism and extensive polyphase deformation and late shearing (Stowe, 1983 and Geringer, 1994). This leads to the proposal that the figure of the conceptual model is not a section view, but a plan view for the Areachap area. If this plan view is taken and shearing is added to it, it becomes obvious that target generation may be very difficult (Stowe, 1983).

3.2.2 Tuffaceous Exhalites

'Tetsusekiei' (Kuroko) and the Main Contact Tuff (Noranda) are examples of tuffaceous exhalites occurring on the seabed over a wide lateral area of smokers (Kalogeropoulus and Scott, 1989). Whether these rocks are truly tuffs needs to be questioned, but that there are chemical sediments related to the black smoker process has been observed at the active black smokers (Binns and Scott, 1993). These cherty exhalites 'host' the massive sulphide lenses. Composition can be variable from hematitic (true Kuroko) to pyrrhotitic (Noranda). Silica (sericitic chert) is very common. Silica facies, through to sulphide (pyrite) to carbonate-facies (dolomite) banded iron stone formations are known (Franklin *et al.*, 1981). Thickness varies from millimetre to one-metre proportions in undeformed situations; they are therefore difficult to detect by conventional mapping and may easily be 'missed'. Massive pyrite 'blows' may occur; these being a frequent source of (geophysical, gossans and geochemical) 'red herrings' (Large *et al.*, 2001).

The exhalative horizon comprises an important element in the model geometry mentioned above. The segment forms one of the long arms of a tipped-over letter 'H' (Large *et al.*, 2001).

3.2.3 Ore Mineralogy and Geochemistry

The main ore minerals are sphalerite, chalcopyrite and galena. Pyrite and pyrrhotite (magnetic) are the dominant sulphides with which the ore minerals are associated. Generally the ore contains more Zn than Cu, percentage wise. The orebody can vary from massive, more than 60 % sulphides to stringer ore, which may vary from disseminated to semi-massive in texture.

The by-products: Ag, Au, Cd (Sn, Se, Te and Bi) and also Zn spinel (gahnite) and cassiterite may be important in exploration (in discriminating barren pyritic zones from massive base-metal sulphides in gossans and stream sediments) (Theart, 1985). Barite, gypsum or anhydrite are present in some younger deposits for example the Prieska Zn-Cu mine where rhodonite has been reported (Theart, 1985 and Wagener and Van Schalkwyk, 1986).

Zonation: Massive, banded sphalerite-pyrite-rich ore (± pyrrhotite) often overlies chalcopyrite-rich massive (± magnetite) and stringer ore. Higher Au and Ag contents are present in the stringer zones (Lydon, 1988).

Pyrrhotite, galena and sphalerite are deformed during deformation, but pyrite tends to remain brittle (Vokes, 1969). A rough positive correlation exists between sulphide granularity and metamorphic grade (Lydon, 1988).

The detection of a Hg halo around Millenbach partially aided in its discovery by lithogeochemical sampling of regional and core samples (Riverin *et al.*, 1980).

3.2.4 Mineralogy and Metamorphism of Alteration Zones

"Rock/water interaction during sub-seafloor geothermal activity causes the congruent dissolution of many major and trace elements from volcanic piles, their transport by hydrothermal fluids and their subsequent precipitation in similar ratios in often more felsic successions". "Cumulatively, this results in a compositional gap in a typical basalt-rhyolite suite being spanned by a continuum of altered rock compositions" (Lydon, 1988). This is reflected by changes in gross mineral paragenesis in the footwall successions hosting the VHMS deposits (Large, 1992).

Most host successions to VHMS deposits have undergone stages and grades of metamorphism as a result of their occurrence in plate-margin (end consumptive–beginning extensional/accretive) zones.

Chlorite, sericite, albite, carbonate, anthophyllite, staurolite, andalusite, kyanite, sillimanite, garnet and cordierite are commonly indicative of metamorphosed alteration zones in volcanic and volcaniclastic successions. Precursor hydrothermal 'conditioning' is responsible for the predominance of seemingly aberrant aluminous silicate mineralogy (more typical of metapelitic rocks) in volcanic rocks of basaltic to rhyolitic composition – even at low metamorphic grades (Large *et al.*, 2001). At the Prieska Cu Zn deposit, it is suggested that the gedrite bearing felses (comprising in increasing order of abundance: tourmaline, phlogopite, albite and gedrite) represent the metamorphic equivalent of the Mg-Fe rich chlorite core of the alteration pipe (Theart, 1985).

With a trained eye, alteration mineral suites can be recognised in the field or in hand specimens, though this usually requires tedious petrographic investigations.

The interpretation of lithogeochemical sampling and whole rock analysis of major and selected minor and trace elements is a much quicker method of confirming alteration signatures in hydrothermal plumbing systems related to massive sulphide deposition.

3.3 LITHOGEOCHEMICAL ALTERATION CHARACTERISTICS AS A VHMS EXPLORATION TOOL

Much of the contribution to an understanding of the nature of the footwall alteration was based on the Millenbach deposit, initially, and since extended to a working model for all VHMS deposits (Riverin *et al.*, 1980; Kalogeropoulus and Scott, 1989 and Knuckey et al., 1982).

3.3.1 Cross-Cutting Footwall Alteration

Although contact metamorphism associated with a subvolcanic granodiorite intrusion has resulted in a typical rock (e.g. 'dalmatianite' - conspicuous cordierite/anthophyllite clots impart a spotted texture), normative calculations show that the footwall alteration pipe, developed below the Millenbach massive ore, initially had chloritic cores and sericitic margins (Riverin *et al.*, 1980). A decrease in Na_2O and CaO and increases in Fe_2O_3 (total iron) and MgO contents are now recognised as typical in alteration pipes immediately below the ore lens. This signature was also found at the Prieska Cu Zn deposit (Theart, 1995). It forms the shorter arm of the tipped-over letter 'H' in the geometric model mentioned above (Large *et al.*, 2001 and Franklin, 1981).

Detailed lithogeochemical studies of various Superior Province greenstone belts concluded that Na_2O and CaO depletion and Fe_2O_3 and MgO enrichment are the most typical trends in the crosscutting footwall of economic sulphide deposits (Cain, 1994 and Gemmell *et al.*, 1992).

This crosscutting soda-depleted alteration zone can extend far downwards stratigraphically below stringer ore, on average up to 1 km. The base of the tapering feature may mushroom out into an area characterised by soda enrichment associated with chlorite and albite alteration (Large, 1992).

Na_20 content is the most useful indicator of alteration as background levels show little variation with degree of differentiation (2.9-3.8%). A Na_2O content of less than 1% invariably indicates footwall alteration (Large, 1992).

3.3.2 Hanging Wall Alteration

The hanging wall is commonly lacking sulphides and alteration is of a very low intensity compared to footwall alteration (Large, 1992).

Large (1992) summarised the hanging wall alteration as being sericite- and chlorite-bearing alteration showing the same chemical trends as the footwall zones, but to a lesser degree. An example at Mount Chalmers showed that Na_2O decreases whereas MgO and the alteration index increase, as the ore horizon is approached from the hanging wall side.

Rare earth element *(REE)* patterns of altered volcanic may change in hanging wall alteration zones, with negative Eu anomalies and low Zr/Y ratios compared to unaltered volcanics (Large, 1992).

3.3.3 Regional Conformable Alteration

The presences of very extensive stratigraphically conformable regional alteration zones are typical of VHMS deposits. These are related to the presence of aquifer, cap rock, recharge channel, discharge channel and heat source (intrusions) as favourable criteria in mega-cell hydrothermal plumbing systems (Large, 1992).

Extensive zones of soda depletion well into the footwall of the Sturgeon Lake - Mattabi and Confederation Lake deposits probably represent aquifers, which were leached with respect to Na_2O, CaO, MnO and Zn by hydrothermal fluids, as a typical example (Large *et al.*, 2001).

The conformable alteration zones are always sub parallel to the exhalites of Segment A, and they form the other long arm of a tipped-over letter 'H', shown as Segment C in the overlay to Figure 8. The conformable alteration zones seem to average ±1 km or so below ore-bearing exhalative zone (Large, 1992).

Care needs to be taken with the 'H' model, when interpreting patterns of soda depletion, to distinguish discharge channels (cross-cutting alteration pipes in the immediate footwall of VHMS deposits) from regional conformable alteration zones sub parallel to the stratigraphy (aquifers). This is particularly the case in strongly tectonised terrain where 'rodding' and/or 'pencilling' can occur to total transposition in extreme cases (probably in the Murchison belt where the 'H' becomes totally flat according to Cain, 1994).

4 REGIONAL GEOLOGY

Outcrop is good in the bulk of the project area, except for areas in and around alluvium-filled washouts or streambeds. In the north, outcrop diminishes beneath the sand of the Kalahari Group and, towards the south, near Copperton; the sediments of the Karoo Supergroup cover the Areachap Group.

4.1 STRATIGRAPHY

4.1.1 Introduction

The mid-Proterozoic Areachap Group consists of varying proportions of amphibolite, hornblende gneiss, quartzo-feldspathic gneiss, calc-silicate and pelitic schist's (Geraghty *et al.,* 1996). It forms a prominent sequence of supracrustal rock types between the eastern margin of the Namaqua Province and the older Kheis Subprovince (Figure 4). The Areachap Group's importance lies in the base metal sulphide deposits within the 250 km outcrop length, and Its fossil meta-island arc character (Geringer *et al.,* 1994).

The stratigraphy of the Areachap Group is given in Table 3 (Geringer *et al.,* 1994). Finality has not been reached with regards to the stratigraphic status of the Areachap Group, but the suggested succession presented in the table below is used for the purpose of this treatise.

Table 3. Lithostratigraphy of the Areachap Group and Relative Equivalence of the Upington, Klein Begin, Boksputs and Copperton Subprojects adapted on Geringer (1994).

<table>
<tr><th colspan="2">Upington and Klein Begin Subproject</th><th>Boksputs Subproject</th><th>Copperton Subproject</th></tr>
<tr><td colspan="4">Namaqua Province (Keimoes Suite Granite)</td></tr>
<tr><td colspan="4">>>>>>>>>>>>>>>>>>>>> Discordant Contact >>>>>>>>>>>>>>>>>>>></td></tr>
<tr><td rowspan="4">AREACHAP GROUP</td><td>Bethesda Formation</td><td>Jacomynspan Formation
Van Wyks Pan Formation
Kantienpan Formation</td><td></td></tr>
<tr><td>Jannelsepan Formation:
Unit 1
Unit 2
Unit 3
Unit 4
Unit 5
Unit 6</td><td>Boksputs Formation:
Unit 1
Unit 2
Unit 3
Unit 4
Unit 5
Unit 6</td><td>Copperton Formation:
Vogelstuisbult Member
Prieska Copper Mines Member
Smouspan Gneiss Member</td></tr>
<tr><td colspan="3">>>>>>>>>>>>>>>>>>>>>> Faulted >>>>>>>>>>>>>>>>>>>>></td></tr>
<tr><td colspan="2">Sprigg Formation</td><td>Eyerdop Pan Formation</td></tr>
<tr><td colspan="4">>>>>>>>>>>>>>>>>>>>>>>> Faulted >>>>>>>>>>>>>>>>>>>>>>></td></tr>
<tr><td colspan="4">Kheis Subprovince</td></tr>
</table>

4.1.2 The Upington and Klein Begin Subprojects

The exploration activities in the Upington and Klein Begin subprojects are concentrated on the Jannelsepan, Sprigg and Bethesda Formations. These formations are grouped together as the Areachap Group on the grounds of:

their geographical position relative to the boundary with the older (2 000 Ma) Kheis Subprovince, their conformable character, their relationship with the late tectonic granites (Keimoes Granite Suite), and the shearing in the area (Geringer *et al.*, 1994).

4.1.2.1 Sprigg Formation

The Sprigg Formation was originally defined by Stowe (1979) as a thin sliver of schist between the Dagbreek Formation (Kheis Subprovince) and Jannelsepan Formation, which comprises a series of outcrops limited to the eastern part of the Areachap Group (Figure 3). The Sprigg Formation contains a predominantly micaceous schist (locally pelitic), including quartzo-feldspathic gneisses, quartzites, amphibolite and a quartzitic conglomerate in the south (Geringer *et al.,* 1994).

The Sprigg Formation's contact with the older immature quartzites of the Dagbreek Formation (Kheis Subprovince) to the east is fault-bounded. An autochthonous origin for the Sprigg Formation is derived from the easterly provenance for the conglomerate clasts.

Faulted contacts between the Sprigg Formation and Jannelsepan Formation are suggested by Vajner *et al.*, (1980), Humphreys (1985), and Pretorius (1986), while Geological Survey maps (Moen, 1988) show intercalation between the Sprigg and Jannelsepan Formation. Poor exposure and lack of contact definition preclude a definite solution to this contradiction.

For the purpose of this text, the Sprigg Formation is regarded as a separate lithological unit from the Jannelsepan Formation on the grounds that the Sprigg Formation could have been derived from the quartzitic rocks east of the Brakbosch-Copperton fault (Figure 4) (Geringer *et al.*, 1994).

4.1.2.2 Jannelsepan Formation

The predominantly meta-volcanic Jannelsepan Formation is the most important unit of the Areachap Group in terms of VHMS exploration and is bounded by shear zones and numerous intrusions of the Keimoes Suite granite of the Namaqua Province (including Straussburg and Louisvale granite).

The Jannelsepan Formation is essentially a succession of metavolcanic and reworked volcanic rocks subdivided into six members (Table 3).

Unit 1 is a quartz-feldspar gneiss comprising intercalated layers and lenses of quartz-feldspar-muscovite gneiss/schist, biotite-muscovite schist and coarse-grained quartz-feldspar-muscovite gneiss with hornblende. Epidote and almandine garnets are occasional accessory minerals. Unit 1 is plausibly derived from felsic volcanics of rhyodacitic composition (Geringer *et al.*, 1987).

Unit 2 consists of massive amphibolites (hornblende) intercalated with quartz-feldspar gneiss (Humphreys, 1985).

Unit 3 comprises feldspathic amphibole gneiss and chloritized feldspathic gneiss.

Unit 4 occurs as lenses of ferruginous amphibole schist and ferruginous chert. Malachite staining is present in the amphibole gneiss.

Unit 5 is described by Cilliers (1987) as a sequence of alternating layers and lenses of homogenous, fine-grained amphibolite, dark-grey medium-grained amphibolite, schistose black amphibolite, calc-silicate-bearing amphibolite and thinly layered calc-silicate and amphibolite.

Unit 6 comprises lenses of black-banded chert and black chert (BIF), which occur on the western flank of the Kraalkop antiform intercalated with ferruginous muscovite schist.

Unit 1 is alternatively known as the Skietbaan Member, while Unit 2 is often referred to as the Donkerspruit Member. Unit 3 and 4 are often grouped together as the Quarry Member, while unit 5 and 6 combined are known as the Swartkop Member (Swartz, 1987).

4.1.2.3 Bethesda Formation

The rocks of the Jannelsepan Formation are, on the west, juxtaposed against the Bethesda Formation. The Bethesda Formation consists of biotite-muscovite schist and metapelitic gneisses. These gneisses were regarded earlier by Vajner (1978) as part of the Jannelsepan Formation. The contact between the Jannelsepan Formation and the Bethesda Formation is obscured by the intrusion of the Keimoes Suite (Louisvale) granite of the Namaqua Province.

Towards the west, the relationship between the Bethesda Formation and the gneisses of the Korannaland Sequence (Namaqua Province) is also obliterated by the intrusion of the late-tectonic Keimoes Suite granites.

4.1.3 Boksputs Subproject

The lithological sequences in the Boksputs subproject are broadly equivalent to those found in the Upington subproject (Table 1), but the sequences represent a separate volcanic centre (Geringer, 1994).

4.1.3.1 Sprigg Formation

See paragraph 4.1.2.1.

4.1.3.2 Boksputs Formation (Jannelsepan Formation)

The Boksputs Formation (Hartebeestpan Formation of Humphreys, 1985) is a lateral equivalent of the Jannelsepan Formation in the Upington area, but represents a separate volcanic centre (Geringer, 1994).

4.1.3.3 Kantienpan Formation

The Kantienpan Formation (Geringer *et al.*, 1987) is equivalent to the Upper Hartebeestpan Formation of Humphreys (1985).

Rocks of the Kantienpan Formation, bounded to the east by the Rooiputs shear, occur mainly in the western portion on the farm Kantienpan and in the northern portion on the farm Van Wyks Pan. These dark, often glassy rocks, with or without garnet, consist mainly of aluminous pelitic gneisses and are intercalated with amphibole gneisses, garnet amphibole gneisses and amphibolites (Geringer, 1994).

4.1.3.4 Van Wyks Pan Formation

This formation consists mainly of granitic-biotite gneisses and quartz-feldspar gneisses intercalated by thin layers of amphibole gneisses (containing garnet and biotite) and amphibolites (Humphreys, 1985).

4.1.3.5 Jacomynspan Formation (Bethesda Formation)

South of the Boven Rugzeer shear zone (Figure 4) the country rock consists of a monotonous sequence of porphyroblastic quartz-feldspar-biotite-garnet gneiss containing minor meta-diorite and amphibolite lenses. Further to the south on the farm Rokoptel, pink gneisses, ranging from fine-grained to coarse pegmatitic or porphyritic muscovite or biotite-muscovite gneiss, often rich in garnets, occur. Differentiated mafic intrusive rocks, such as norite, harzburgite and pyroxenite, grading into anorthositic gabbro, occur as sheet-like structures or dykes. These mafic intrusions form the host rock for low-grade nickel-copper mineralisation on the farms Jacomynspan and Rokoptel (Attridge, 1986).

4.1.4 Copperton Subproject

The meta-volcano-sedimentary rocks in the Copperton Volcanic Centre are broad equivalents of those in the Upington Volcanic Centre.

4.1.4.1 Eyerdop Pan Formation

The Eyerdop Pan Formation is tectonically in the same position as the Sprigg Formation (see paragraph 4.1.2.1) of the Upington Subproject Area and consists of 150 m thick conglomerate within a schistose matrix (Humphreys, 1985).

4.1.4.2 Copperton Formation

The meta-volcano-sedimentary sequences in the Copperton and Kielder region are recognised as lateral equivalents of the Jannelsepan Formation (Upington Subproject) and the Boksputs Formation (Boksputs Subproject), owing to its geochemical composition and tectonic history (Humphreys, 1985).

The Copperton Formation consists of essentially of three main members, which can be correlated with the six units in the Jannelsepan and Boksputs Formations (Theart, 1985 and Theart *et al.*, 1989). The three members are summarized by Theart *et al.* (1989) and are as following:

- The Smouspan Gneiss Member is a metaluminous, homogeneous gneiss, which consists mainly as a biotite-hornblende-quartz-plagioclase gneiss. This Member is considered to form the base of the Copperton Formation and is believed to have originated as a dacitic lava.
- The Prieska Copper Mines Member consists of peraluminous silicate rocks and hosts the massive sulphide ore of the Prieska Zn-Cu mine. The contact with the Smouspan Gneiss Member and this Member is gradational, so also is the contacts between the different rock types within this Member. The Prieska Copper Mines Member mainly consist of a gedrite fels, the hydrothermally altered dacite protolith of the Smouspan Gneiss Member, and a quartz-perthite-sillimanite gneiss, originating from precipitation of

silica, sericite and trace minerals in basin-floor sediments close to the fumarolic vent.

- The Vogelstruisbult Member consists of a tholeiitic banded hornblende gneiss of intermediate composition, a laminated amphibolite, originating from a basaltic lava or subaqueous tuff, and metapelites originating from immature sediments derived from basaltic to intermediate volcanic rocks. The Vogelstruisbult Member is thought not to reflect the ore-forming hydrothermal alteration and is considered to lie stratigraphically above the Prieska Copper Mines Member.

Massive amphibolite layers, believed to have a tholeiitic lava as precursors, randomly occurs throughout the Copperton Formation and are not considered to be part of the original layered sequence (Gorton, 1981). Theart (1985) concluded that the massive amphibolite layers probably intruded the sequence as dykes or sills after the ore formation, but prior to the regional deformation and metamorphism.

4.2 STRUCTURE

4.2.1 Introduction

The Areachap Group extends across two tectonic provinces, the Kheis Subprovince and the Namaqua Metamorphic Province. The Kheis Subprovince is also known as the Eastern Marginal Zone of the Namaqua Province (Humphreys, 1985 and Stowe, 1983). Joubert (1986) suggested that the Namaqua Province should be divided in a Gordonia Subproince in the east and the Bushmaland Subprovince in the west, separated by the Hartebeest River thrust, which have a northwest-southeast (Figure 4). Stowe (1986) describes the Gordonia Subprovince as a microcontinent-arc complex that was accreted during the early stages (1300 Ma) of the Namaqua orogeny.

The Brakbosch fault, in the east, and the Bovenrugseer shear zone, in the west, defines the Areachap terrane. The Trooilapspan or Koegrabe shear zone distinguishes between the Jannelsepan Formation to the west and ultramafic rock to the east, which are not part of the Areachap Group. The Areachap Group thins towards the south-east where the Bovenrugseer shear and Brakbosch fault merge (Prinsloo, 1998).

Stratigraphic contacts between the different formations are not easily defined accurately, owing to complex structural textures. Ludick (1987) found that the contact between the Jannelsepan and Bethesda Formations are obscured by shearing, implying that the Bethesda Formation does not necessarily post date the Jannelsepan Formation. However, Geringer (1983) observed from borehole data that the Bethesda metapelites might overlie the Jannelsepan Formation stratigraphically.

4.2.2 Faulting and Folding

The Areachap Group has been greatly affected by tectonic events (Table 4) since it represents an island arc deposit sandwiched on the east by the Kheis Subprovince (the eastern margin of the Kaapvaal Craton) and on the west by the Namaqua Province. A number of smaller and larger splay faults form a network of mainly NW-SE-trending faults and shears across the Areachap Group. This network of faults divides the area up into sub-parallel shear domains (Stowe, 1983 and Humphreys, 1985).

Very prominent faulting occurs throughout the Areachap Group. The Brakbosch Fault, which can be traced from satellite images for 220 km from north of Upington to south of Copperton, forms the boundary of the Areachap Group with the Kheis Subprovince (Figure 4). Other large faults and shear zones, like the Boven Rugzeer shear zone, play an important role in the structural setting of individual formations in the Areachap Group. Shears and faults of lesser significance are common (Stowe, 1983 and Humphreys, 1985).

Table 4. Characteristics of Deformation Phases in the Kheis Subprovince and Namaqua Provinces

	Kheis Subprovince	**Namaqua Province**
F1	- F1a Isoclinal, SE vergence NNE trend. - F1b Isoclinal N to NE trend.	Only one phase distinguished; isoclinal to tight, yielding widespread S1 foliation.
F2	NNW-trending tight folds producing strong L2 in quartzites.	NNW-trending closed to tight folds commonly with an L2 lineation and developing rare S2 axial planar foliation.
F3	Frequently box-folds on mesoscopic scale. Not as well-developed as F2; attitude uncertain.	ENE- to NE-trending open folds creating dome-and-basin terrain with F2; frequently undulating.
F4	Rare monoclinal structures.	Macroscopic structures, near co-axial with F2, causing steepening of western limbs of F2 antiforms and loss of intervening synforms.
F5	Faults.	Shears and faults.

The deformation phases to which the Areachap Group are subjected are (Stowe, 1983 and Humphreys, 1985):

- Early deformation (F1) imparts a strong penetrative foliation (S1) in the Namaqua Province. This deformation gradually weakens eastward, so that in the Kheis fold-and-thrust belt the F1 it is only penetrative in schistose lithologies. At the Prieska Zn-Cu deposit the earliest deformation seen as isoclinal folds, originally with subhorizontal coaxial planes may belong to this phase (Theart, 1985).
- F2 deformation, directed initially from the south, is expressed as NNW-trending closed syn- and antiforms. It has an almost upright axial planar attitude with slightly WSW strike. The earlier isoclinal fold at the Prieska Zn-Cu deposit was deformed in tight folds with a vertical axial plane that may be correlated with this event (Theart, 1985).
- The wavy F3 deformation, possibly the result of slip on contemporary conjugate shears, varies in trend from E-W in the north to NE-SW in the south, and forms a dome-and-basin pattern with F2. This is also seem at the

Prieska Zn-Cu mine, as isoclinal folds with subhorizontal coaxial planes that were refolded by subsequent F3 and F4 interference dome structures (Theart, 1985).

- F4 is represented by large-scale folds, which steepen and shear the westerly F2 antiform limbs.
- F5, encompassing a distinctly earlier (F5a) and later (F5b) deformation stage, culminates in strike-slip activity which commences during the F2 stage. The faulting is predominantly right-lateral and trends NNE to NW.Joint analysis of the F5a phase of deformation shows that the maximum principal stress is directed NE-SW. The undulation in the postulated stress field suggests that F5b faulting is responsible for a degree of rotation, though this cannot be directly proved or evaluated.

The VHMS conceptual model relies heavily upon recognising the respective geometry of footwall and hanging wall alteration. An understanding of the folding is vital to the VHMS model, since overturned beds owing to overfolding are a common occurrence in the Areachap Group. This complexity has resulted in directly opposing interpretations by Wagner and Van Schalkwyk (1986) and Theart (1985) and Theart *et al.* (1989).

The Areachap Group is situated on the eastern margin of the Namaqualand Mobile Belt which is characterised by the interference of two tectonic environments: the Kaapvaal Craton and, more specifically, the Kheis Subprovince to the east, and the later, more discernible Namaqua Province evolution to the west (Geringer, 1994).

The boundaries of the different provinces are well defined by geological and geophysical constraints and feature as distinct lines on geological maps. The peripheral effects of the main tectonic episodes, however, extend beyond such boundaries. Tectonic effects therefore overlap rather than help to form accentuate natural distinctions between provinces, hence the intense folding found throughout the Areachap Group (Table 4) (Stowe, 1983 and Schalkwyk, 1986).

4.3 GENESIS

4.3.1 Introduction

The major petrographic and geochemical features, as well as the most probable protoliths for the various gneisses of the Areachap Group are given in Appendix F. Rocks of the Areachap Group resemble island-arc related volcanics and volcanic derived sediments (Appendix F) (Geringer, 1994).

4.3.2 The tectonic framework within which the Areachap group formed and its subsequent modification.

Various tectonic models, based on the structural, metamorphic, geochemical, and isotopic character of the Namaqua Province have been proposed. Stowe (1983) and Van Bever Donker (1991) suggest that the structural features of the Namaqua Province can be related to eastwardly directed forces which culminated in subduction and collision of the Namaqua and Kaapvaal Cratons resulting in an island arc environment and the formation of the Areachap Group. Humphreys (1990), however, points out that the long duration of heat flow necessary to cause high temperature/low pressure metamorphism documented in the Areachap Group, is not in favour of a subduction-related model but that hot spots caused the high heat flow in the area.

An alternative possibility for the tectonic evolution of the area is that the entire zone developed in a transtensional structural environment a discussed by Dewey *et al.* (1998) and Krabbendan and Dewey (1998). It is possible that the deformation from at least F3 and onwards, may be related to transtensional deformation.

Current plate tectonic models indicate that calc-alkaline volcanics, as found in the Areachap Group, occur predominantly along destructive plate margins. Calc-alkaline assemblages found in the Areachap Group include low-K arc tholeiite, calc-alkaline basalt, and high-shoshonitic basalt (Geringer, 1994).

Support for the orogenic environment comes from the uranium-lead isotope ratios of the amphibolites (Cilliers, 1987).

Radiometric age determinations on the amphibolites and gneisses indicate that the Areachap volcanic arc formed over a considerable period of time, starting at 1 600 Ma and ending around 1 285 Ma (Cornell *et al.*, 1990). It is difficult to explain the long duration of development in the Areachap volcanic arc by means of a single, continuous process.

Geochemical variations in different areas indicate differences in tectonic environments in which these rocks were deposited.

The low-K tholeiitic, calc-alkaline to high-K, shoshonitic character of the amphibolites from Boksputs to Upington subprojects may either indicate an increase in arc maturity from south to north or it may reflect arc compositional polarity in the same direction (Geringer, 1994).

Chemical variation of the protoliths of the hornblende gneiss and quartz-feldspar gneiss are also reconcilable with arc maturity or polarity, showing an increase in the felsic components from south to north in the area (Geringer, 1994).

Deposition of the Sprigg Formation, with its channel conglomerates, probably took place on a stable shelf back-arc environment but the exact relationship of the Sprigg Formation with the remainder of the Areachap Group is uncertain (Geringer *et al.*, 1994).

The hornblende, biotite, and pelitic gneisses and banded iron formations of the Bethesda, Kantienpan, and Copperton Formations are chemically similar to immature sediments. The biotite gneiss and metapelites may represent products formed from dacitic and rhyodacitic source regions, deposited in fore-arc basins (Geringer *et al.*, 1994).

The large varieties of rocks of the Areachap Group, which are restricted in a defined area, reflect the extremely complex deformational history along the eastern margin of the Namaqua Province. Low pressure/high temperature metamorphism, reconcilable with a subduction related environment reached a peak around 1 200 Ma. Subduction ceased at this point and was transformed into a collision phase. The collision was marked by the emplacement of large volumes of calc-alkaline granite magma of Keimoes Suite (Stowe, 1986 and Geringer *et al.*, 1988). This event was dated at around 1 150 Ma by Barton and Burger (1983).

The tectogenesis ended with large-scale movements along major shear zones (Stowe, 1986) (Table 4) while the last metamorphic effects occurred approximately 943 Ma ago (Cornell *et al.*, 1986).

5 LOCAL GEOLOGY

5.1 LITHOLOGY

5.1.1 Hanging Wall

The hanging wall mostly consists of biotitegneiss, which comprises massive, unlayered rocks made up of patches of coarse-grained quartz and feldspar (plagioclase and microcline) separated by patches of granular fine-grained quartz, plagioclase and microcline. Also present are major to minor amounts of brown biotite, magnetite, and traces of green hornblende and zircon. The majority of opaque minerals consist of interstitial magnetite and pyrrhotite in close association with one another, together with small amounts of pyrite and ilmenite. Ilmenite may occur both as discrete granules, and as coarse lamellae exsolved from a titaniferous magnetite (Richards, 1999).

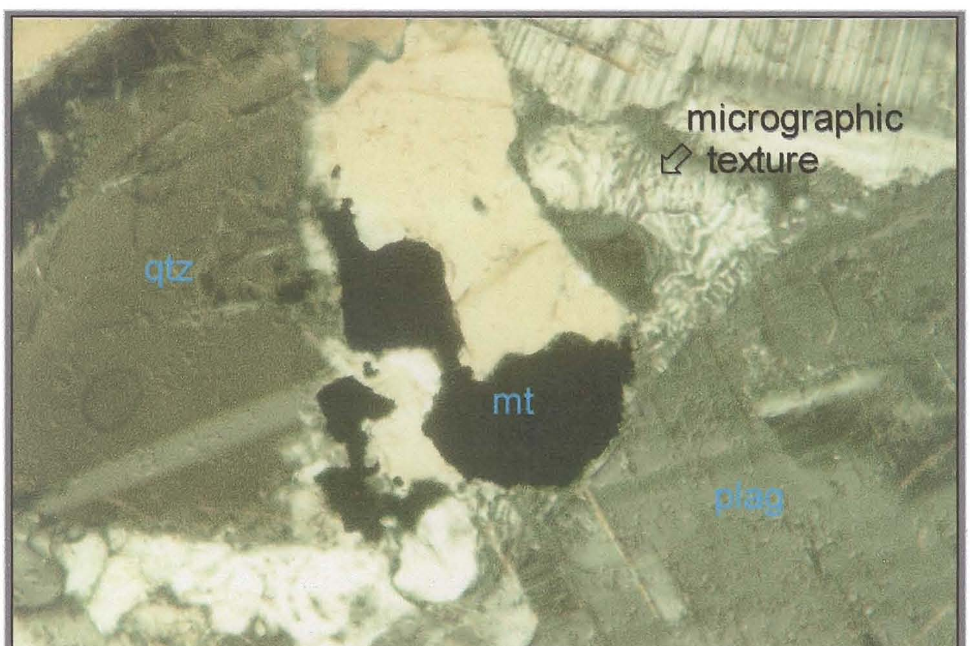

Figure 7. Hanging wall mineralogy. Micrographic texture occurs typically between areas of coarser-grained quartz (qtz) and feldspar and finer-grained intervening areas. (Polarised light, X 100) (Richards,1999). (mt – magnetite and plag – plagioclase).

The composition and textures present in the rock suggest that this was originally a magmatic igneous rock. The presence of interstitial magnetite and ilmenite, green hornblende and zircon, is consistent with a deep-seated magmatic proto-rock of possible granitic composition. The micrographic textures may own to a later hydrothermal event after the initial upper amphibolite metamorphism. The metamorphism appears to have affected the proto-rock with the formation of biotite, the mobilisation and recrystallisation of small amounts of magnetite, and the recrystallisation of some of the finer-grained quartz in particular into elongated subparallel crystals. The rock can probably be best classified as an orthogneiss (Richards, 1999).

5.1.2 Footwall

The footwall consists mainly of biotite hornblende gneiss (Figure 8) and amphibolite, with the amphibolite being possible later intrusive sills. On a microscopic scale, this rock is very poorly foliated. It consists principally of quartz with minor amounts of plagioclase, together with ragged flakes of biotite, which imparts a penetrative foliation to the rocks.

Most biotite, however, is located along grain boundaries. In addition, minor to trace amounts of pyroxene and hornblende are present as irregularly shaped bodies, interstitial to quartz. Zircon is present in trace amounts as inclusions in biotite, hornblende and quartz. In addition, there are traces of garnet in the rock. Opaque mineralisation is present in major to minor amounts and consists predominantly of magnetite, with lesser amounts of pyrite, pyrrhotite and very small amounts of chalcopyrite. The minerals in this gneiss are, in decreasing order: plagioclase, quartz, hornblende, biotiete, orthoclase, sphene, magnetite / ilmenite, apatite and zircon.

The texture of the rock is granular, with a moderately large degree of variation in grain size, being made up of clusters and layers of moderately coarse-grained, interlocking quartz with intervening areas, layers and lenses of fine-grained, interlocking quartz and occasional feldspar. Quartz tends to be greatly elongated in the direction of foliation in coarse-grained layers, and associated finer-grained feldspar commonly contains worm-like inclusions of quartz in a micrographic texture.

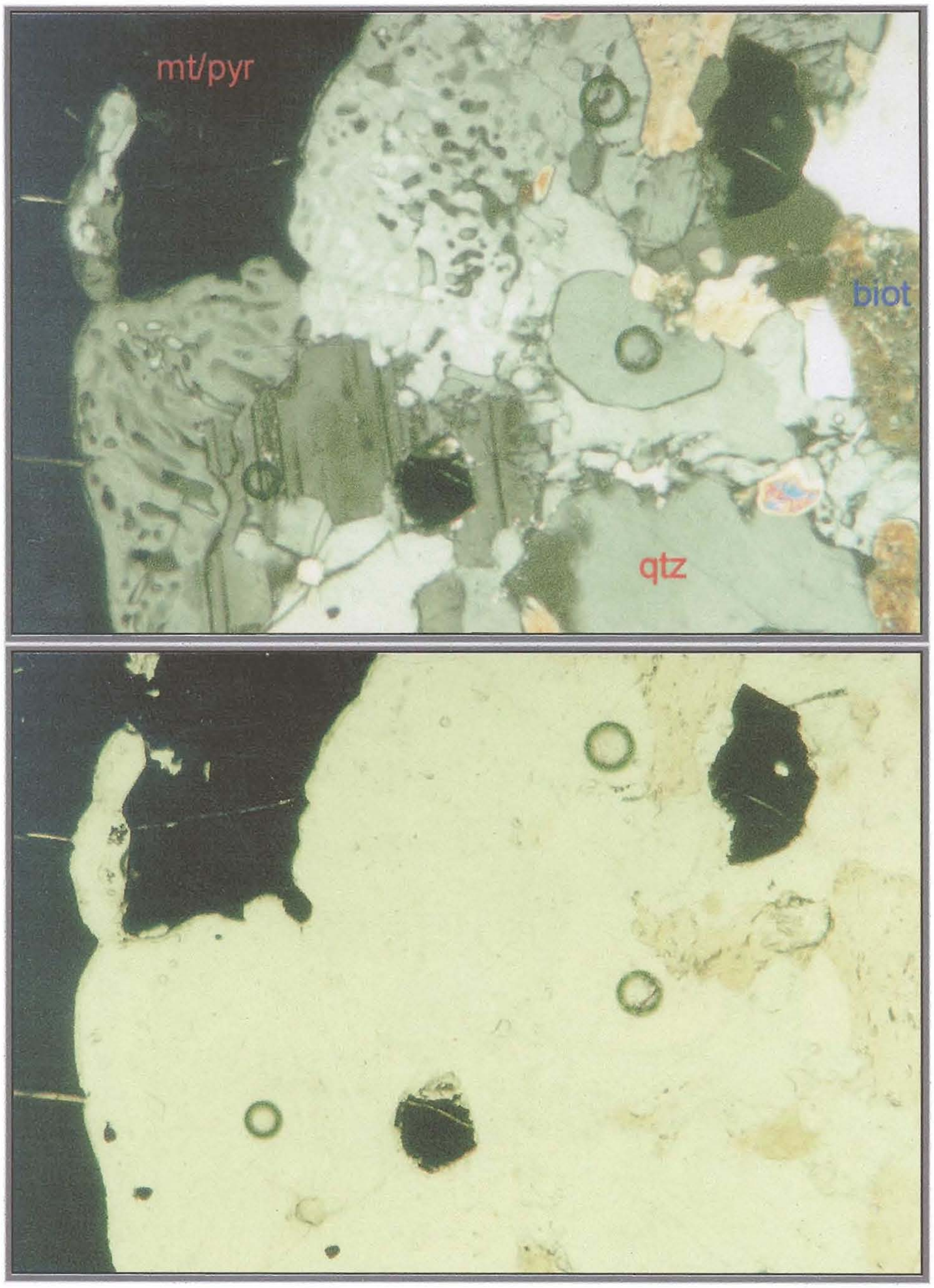

Figure 8. Footwall mineralogy. Micrographic textures between alkali feldspar and quartz (qtz) commonly occur in finer-grained areas that also contain biotite (biot), plagioclase and opaque mineralisation, between coarser-grained quartz and feldspar. (pyr – pyrite and mt – magnetite). (Top: polarised light; bottom: plane light; X 100).

Minor orthopyroxene, with green-pink pleochroism (i.e. hypersthene), are also present. It is generally intergrown with quartz and feldspar as ragged anhedral grains, and appears to be confined to the predominantly fine-grained regions of the rock grains (Richards, 1999). This is indicative of upper amphibolite to granulite facies that existed during the metamorphism.

Compositionally and texturally, the rock is poorly foliated biotite hornblende gneiss that appears to have resulted from the metamorphism of an igneous intrusive rock, possibly granite. The variation in grain size, the presence of micrographic textures, the occurrence of trace amounts of zircon and of relatively large amounts of magnetite suggest that it could have been a granite in which the quartz has subsequently undergone substantial modification to form elongated crystals, as well as foliation. The finer-grained parts of the rock appear to have been affected and modified to a much lesser degree. Also of note is the apparent second orientation direction displayed by magnetite mineralisation, which is presumably stress-induced grains (Richards, 1999).

A massive amphibolite occurs randomly throughout the Jannelsepan Formation, but it is not considered to be part of the original layered sequence, the same as with the massive amphibolite in the Copperton Formation (Theart, 1985). This hornblende-plagioclase amphibolite is believed to have a intrusive basaltic rock as precursors, owing to the rather featureless and uniform texture and grain size which are characteristic of sills and dykes. The conclusion was the same as in the case with the Copperton Formation, that the massive amphibolite layers probably intruded the sequence as dykes or sills after the ore formation, but prior to the regional deformation and metamorphism.

5.1.3 Sulphide Zone

Samples were taken from a less sulphide-rich part of the core, but with visible chalcopyrite in the hand specimen, as well as from an area of massive sulphide mineralisation. The main sulphide minerals present are pyrrhotite and sphalerite, with lesser amounts of chalcopyrite and pyrite. Galena is present in trace amounts only (Richards, 1998).

Sulphide mineralisation may occur as either patches of massive mineralisation or disseminated in the intervening gangue minerals of the host rock. In areas of massive mineralisation, pyrrhotite and sphalerite appear to be particularly well associated with each another. Chalcopyrite is usually present in only trace amounts and seems to occur as relatively isolated 'patches' throughout the host rock mineralisation. Massive pyrrhotite/sphalerite appears to replace the protolith almost completely, but a relatively large number of remnants, generally subrounded particles of host rock, occur within the pyrrhotite and sphalerite as inclusions. These clasts are interpreted as the product of "durchbewegung" (Theart, 1995).

There is no evidence of grain boundaries in either massive pyrrhotite or massive sphalerite, but both minerals have been extensively fractured in places, particularly pyrrhotite, and the microfractures were infilled probably with chlorite. The fracturing can be a product of later effects. Pyrrhotite and sphalerite are generally coarsely intergrown with one another and small amounts of chalcopyrite may also be incorporated into the texture. Chalcopyrite, when present, generally occurs along, or adjacent to, the common boundary between pyrrhotite and sphalerite as an exolution feature (Richards, 1998).

Small inclusions of sphalerite may be present in pyrrhotite, but are rare; size is usually <100 μm. and commonly 20 – 30 μm. Small subrounded inclusions of pyrrhotite may be found in sphalerite and are more common than sphalerite inclusions in pyrrhotite. The pyrrhotite grain size is generally <20 – 30 μm with occasional larger inclusions up to 50 μm in diameter. Inclusions of chalcopyrite in sphalerite also occur and are common in areas where chalcopyrite is associated with the common boundary between sphalerite and pyrrhotite, but are less common elsewhere. The chalcopyrite grain size is generally <20 μm. Very fine-grained, almost submicroscopic, linear bodies of exsolved chalcopyrite are found in sphalerite, but are generally widely dispersed and relatively rare (Richards, 1998). This exsolved chalcopyrite is also common at Prieska Cu Zn mine.

Coarser-grained chalcopyrite is usually found closely associated with the common boundary between pyrrhotite and sphalerite, and is generally of irregular shape and of variable grain size. Width of these grains can vary from <10 μm to >200 μm. Chalcopyrite may contain a small number of inclusions of sphalerite and pyrrhotite of mostly less than 35 μm in size, while the shape of inclusions is largely irregular, to subrounded (Richards, 1998).

Pyrite occurs as small to large, irregularly-shaped bodies which apparently replace larger masses of sphalerite pyrrhotite and chalcopyrite. The pyrite can contain small inclusions of other sulphide minerals, and sometimes also gangue. The size of the inclusions is generally <100 μm. The size of the pyrite grains can be greater than 2.5 to 3 mm in diameter.

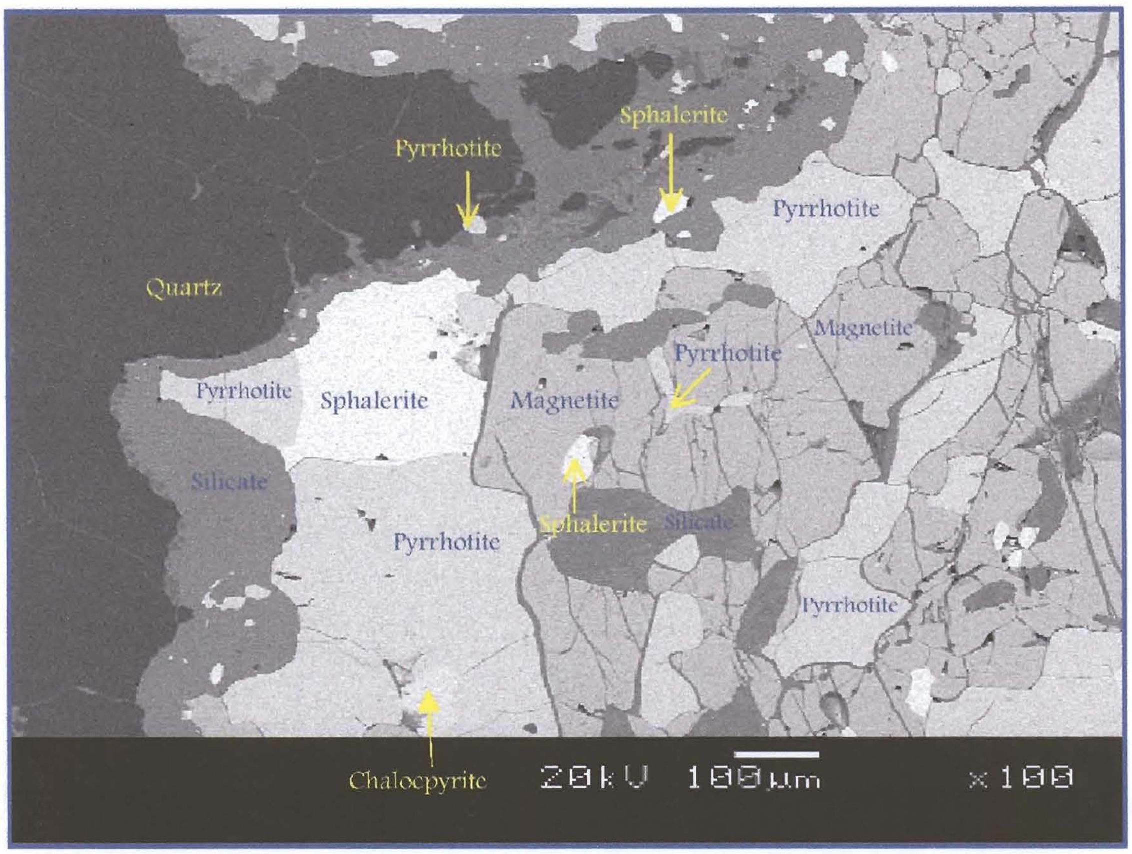

Figure 9. Sulphide mineralogy. Silicate replacing magnetite intergranular, as well as occurring along fractures and margins. Magnetite is also being replaced by pyrrhotite and sphalerite.

Intergranular sphalerite generally appears to contain large numbers of inclusions of chalcopyrite, either as small irregularly-shaped grains of <10p.m in size, or as exsolved lamellae of <2 – 3 µm in width. Chalcopyrite inclusions are far more numerous in this finer-grained sphalerite than in the coarser-grained, more massive sphalerite (Richards, 1998).

Very fine-grained sulphides, consisting predominantly of pyrrhotite and, to a much lesser extent, chalcopyrite and galena, can occur as disseminated particles throughout the host rock, mainly in association with chlorite. Grain shape is irregular (i.e. subangular to subrounded) and size is generally 25 – 75 µm with some particles of up to ±150 µm in size. Quartz in the host rock is generally free of sulphide mineralisation. Only trace amounts of sphalerite are found as fine-grained disseminated particles in the host rock (Richards, 1998).

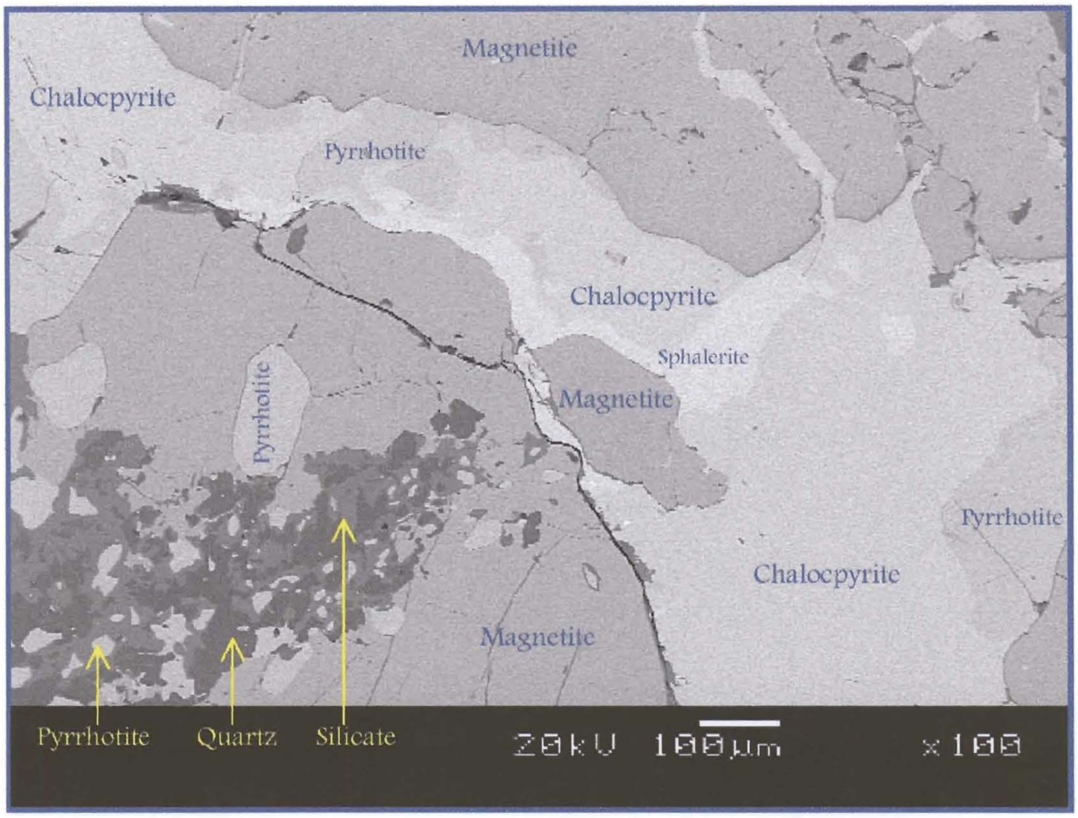

Figure 10: Sulphide mineralogy. Magnetite being invaded by sulphides, that in turn exhibit multiple and consecutive replacements among themselves.

5.2 MICRO-ANALYSES

The compositions of the specific minerals were determined with energy dispersive X-ray (EDS) analysis. The main substitute for Zn in sphalerite is iron (Fe), while manganese (Mn) and cadmium (Cd) are also common, so that sphalerite may be expressed as (Zn, Fe, Mn, Cd) S. Sphalerite in the sample under investigation has an average Fe content of 8.47% (Figure 11), while other sulphide minerals are essentially free of contaminants. Gangue minerals appear to be mainly quartz and silicates, varying in composition between CaFeMg-silicates and CaFe-silicate, like hornblende, anthophyllite and cummingtonite-grunerite (Reyneke, 2002).

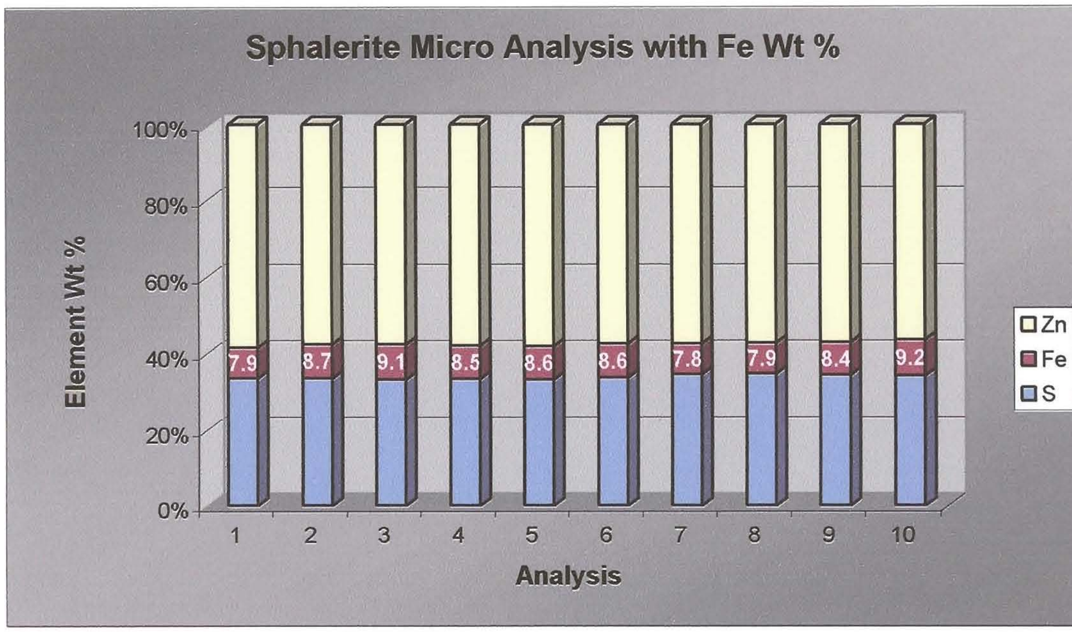

Figure 11: Sphalerite micro-analyses (Reyneke, 2002).

Other samples were also analysed by using electron microprobe analysis with a semi-quantitative program. The results can be seen in Table 5.

Table 5. Electron microprobe analysis in percentage weight of core samples.

Mineral	Fe	Cu	Mn	Zn	S
Sphalerite	7.06	0	0.36	59.16	33.42
	6.89	0	0.32	59.22	33.57
	7.29	0	0.30	58.67	33.73
	7.04	0.02	0.27	59.43	33.24
Pyrrhotite	59.80	0	0.11	0.08	40.01

Note that there is a consistent ±7% (wt) Fe contained within the sphalerite. Fe is the main substitute for zinc in sphalerite, and according to Deer *et al.* (1992), up to 26% (wt) Fe has been reported in sphalerite. For beneficiation purposes, high Fe-bearing sphalerite does not apparently respond as well to the normal flotation process as sphalerite with a low Fe content. The high iron content sphalerite will decrease the plant yield, which means less Zn concentrate and a lower revenue. Detail studies should be conducted, in the pre-feasibility and feasibility stage, to quantify the effect of the high iron content sphalerite in the plant and to determine if new plant designs can't address the possible problem.

5.3 DISCUSSION

The principal sulphide present in the sample is pyrrhotite with a lesser amount of sphalerite. Small amounts of chalcopyrite, pyrite and traces of galena also occur, but are not uniformly distributed throughout the mineralised zone. The host rock minerals appear to consist of quartz with lesser amounts of chlorite. These statements means that drill holes has not intersected any feeders, if it have not been eroded due to the steep dipping of the deposit.

Pyrrhotite and sphalerite occur in massive form, becoming interstitial to host rock gangue minerals in less mineralised areas. Fine-grained sulphides (mainly pyrrhotite with only traces of sphalerite) occur throughout the less well-mineralised parts of the host rock, where they seem to be associated principally with chlorite.

Chalcopyrite occurs in isolated patches throughout the zone of mineralisation and, when present, appears to concentrate along common boundaries between pyrrhotite and sphalerite. Pyrite can replace both pyrrhotite and sphalerite, but is not widely distributed through the mineralised zone.

Massive and coarse-grained sphalerite contains a number of inclusions of both pyrrhotite and chalcopyrite; these are generally <20 – 30 μm in size. A small amount of chalcopyrite occurs as extremely fine-grained exsolved blebs and laminae in the sphalerite.

The relatively large grain size and massive to interstitial habit of much of the ore indicates that it should be possible to produce a high grade concentrate with high recovery rates, using routine beneficiation procedures for a sulphidic ore of this type. It should, however, be borne in mind that large amounts of pyrrhotite are present which will have to be separated from sphalerite and chalcopyrite. Removal of pyrite will probably be quite easily achieved because of its relatively large grain size, smooth grain boundaries and comparatively isolated granular habit. Pyrrhotite on the other hand can be separated magnetically and can be dispose of more easily that the pyrite.

It should be noted that a sphalerite concentrate would contain a certain amount of chalcopyrite as a result of the presence of exsolved chalcopyrite in the sphalerite ore. In addition, the presence of chlorite in the ore suggests that large amounts of slimes may be generated in the beneficiation process and may have to be suppressed.

Electron-microprobe analyses of sphalerite indicate that approximately 7% (wt) Fe may be expected to occur in all the sphalerite. The high Fe percentage may be enough to affect the floatation of sphalerite adversely and should be investigated further (Richards, 1998). It is proposed that pilot floatation test work be done at Rosh Pinah with a bulk sample of the high Fe-sphalerite.

6 THE KANTIENPAN MASSIVE SULPHIDE BODIES

6.1 MODELLING

The deposit was modelled using the Lynx computer programme, and a representation of the model can be seen in Figure 12. Looking in a westwardly direction, the green represents the surface, blue the main sulphide body, yellow the second sulphide lens and the drill holes are represented by the black lines.

The dimensions of the sulphide body are a length of 800 meters, a width of 300 meters and an average true thickness of 6.01 meters. These dimensions, when multiplied with a relative density of 3.47 g/cm^3, amounts to a resource of 5 Mt.

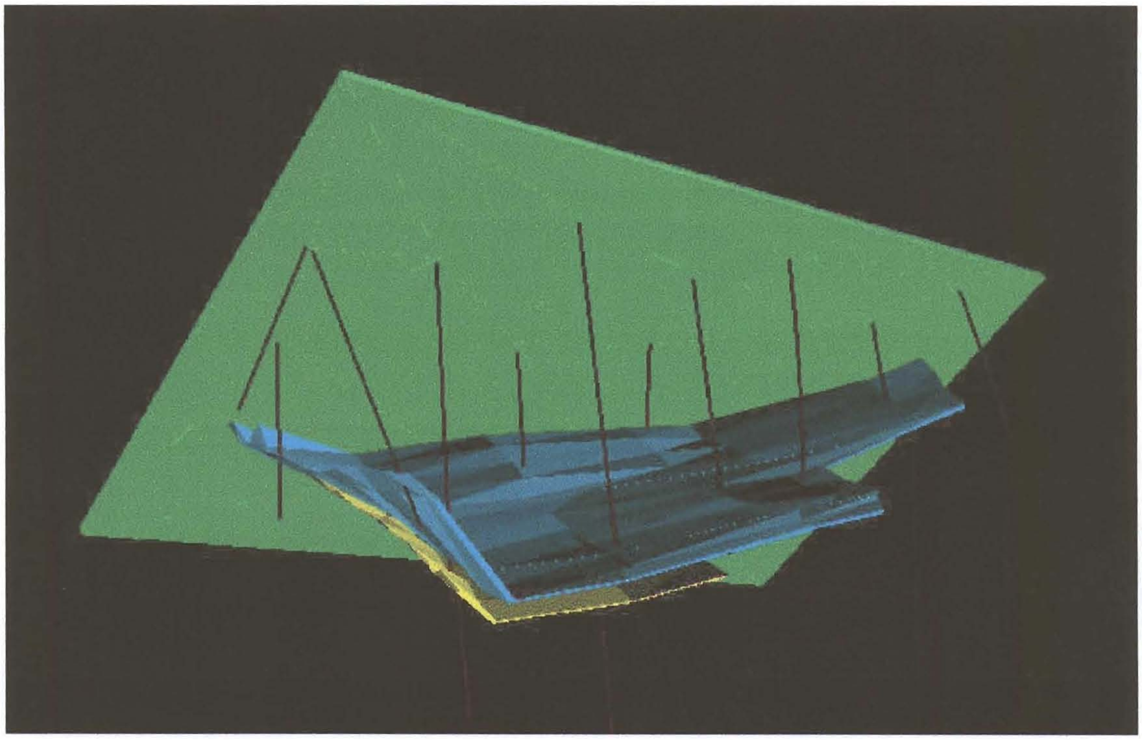

Figure 12. Lynx 3D model.

The Zn, Cu values and the ratio of Zn/Cu were contoured and can be seen in Figure 13, with the highest values to the south of the deposit.

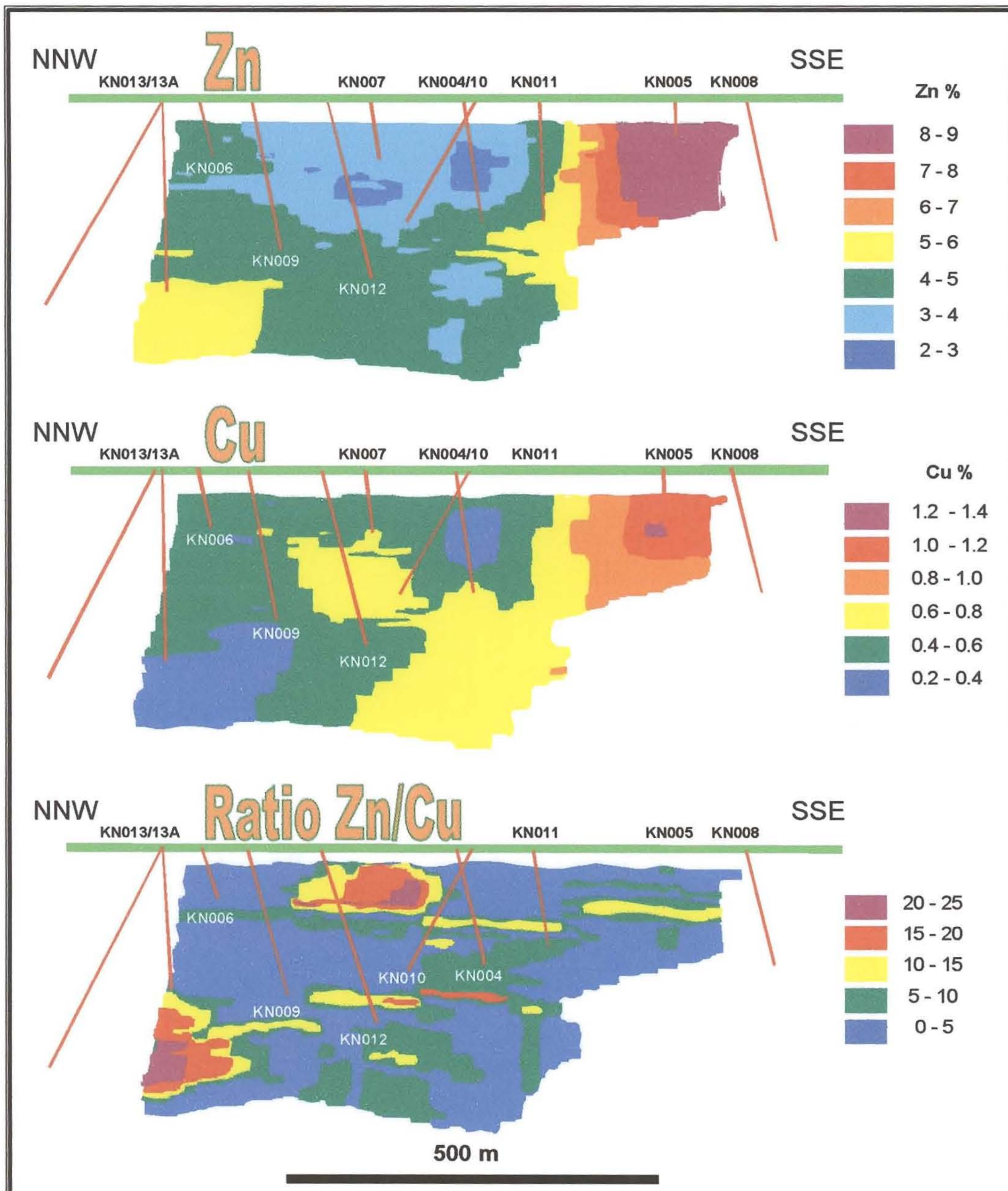

Figure 13. Longitudinal sections with chemical contours.

6.2 GEOSTATISTICAL EVALUATION

The drilling results from the Kantienpan Grid KNP/C were used to evaluate the massive sulphide body found on the farm Kantienpan. A cut-off of 1% Zn was used so as only to take the massive sulphide into account, and not the stringer and disseminated zones. The drilling results are shown in Table 6.

Table 6. Drilling results with a cut-off of 1% Zn.

Borehole no.	Final Depth (m)	CUT-OFF 1% Zn				
		From (m)	To (m)	True Width (m)	Zn%	Cu%
KN001	309.33	No intersection				
KN002	234.03	No intersection				
KN003	302.80	192.00	205.00	13.00	3.96	0.36
KN004	154.89	106.89	115.89	9.00	1.27	0.14
		106.89	109.89	3.00	2.21	0.06
KN005	151.32	82.05	90.89	8.84	6.32	1.02
KN006	140.00	103.69	104.69	1.00	4.59	0.24
KN007	140.00	105.96	112.96	7.00	3.15	0.57
KN008	155.30	No intersection				
KN009	280.80	241.37	243.87	2.50	4.50	0.56
KN010	242.20	190.02	196.17	6.15	4.74	0.49
KN011	239.50	204.07	206.70	2.63	6.59	0.35
KN012	307.15	278.34	281.31	2.97	5.09	0.30
		291.82	292.70	0.88	7.42	0.26
KN013	256.60	Rods stuck – borehole abandoned				
KN013A	284.95	255.21	256.68	1.47	2.57	0.09
		259.00	259.83	0.83	1.29	0.23

All the holes with no massive sulphide intersections were removed and the second body, intersected in boreholes KN012 and KN013, were not taken into account. The intersections in KN004 were used as a whole, thus including some of the waste in the massive sulphide unit. The intersection of KN003 was reduced to give the true thickness, because it was drilled along the dip.

Table 7. Drilling results of only the sulphide intersecting zones.

Borehole no.	Final Depth (m)	CUT-OFF 1% Zn				
		From (m)	To (m)	True Width (m)	Zn%	Cu%
KN003	302.80	192.00	205.00	5.50	3.96	0.36
KN004	154.89	106.89	115.89	9.00	1.27	0.14
KN005	151.32	82.05	90.89	8.84	6.32	1.02
KN006	140.00	103.69	104.69	1.00	4.59	0.24
KN007	140.00	105.96	112.96	7.00	3.15	0.57
KN009	280.80	241.37	243.87	2.50	4.50	0.56
KN010	242.20	190.02	196.17	6.15	4.74	0.49
KN011	239.50	204.07	206.70	2.63	6.59	0.35
KN012	307.15	278.34	281.31	2.97	5.09	0.30
KN013A	284.95	255.21	256.68	1.47	2.57	0.09

Both the Cu and Zn were weighted against the thickness of the intersection and the calculations used from here onwards are based on this assumption.

A Zn equivalent (Zn Eq) was also calculated with a Zn:Cu ratio of 1: 1.75, based on the average daily price of the two commodities from 1989 to 2000 (London Metal Exchange website: http://www.lme.co.uk). The results, including the Zn Equivalent, with a standard error of 0.4, are shown in Table 8. It may be more appropriate to use a 1% Zn Eq cut-off rather than a 1% Zn cut-off, but a 1% Zn cut-off was used during the project phase and it was decided to keep on using this.

Table 8. Width weighted averages and a Zn equivalent.

					Weighted against width		
Borehole	**Width (m)**	**Zn%**	**Cu%**	**Zn Eq %**	**Zn %**	**Cu %**	**Zn Eq %**
KN003	5.50	3.96	0.36	4.59	0.46	0.04	0.54
KN004	9.00	1.27	0.14	1.52	0.24	0.03	0.29
KN005	8.84	6.32	1.02	8.11	1.19	0.19	1.52
KN006	1.00	4.59	0.24	5.01	0.10	0.01	0.11
KN007	7.00	3.15	0.57	4.15	0.47	0.08	0.62
KN009	2.50	4.50	0.56	5.48	0.24	0.03	0.29
KN010	6.15	4.74	0.49	5.60	0.62	0.06	0.73
KN011	2.63	6.59	0.35	7.20	0.37	0.02	0.40
KN012	2.97	5.09	0.30	5.62	0.32	0.02	0.35
KN013A	1.47	2.57	0.09	2.73	0.08	0.00	0.09
Weighted averages					**4.09**	**0.49**	**4.94**

6.3 KANTIENPAN RESOURCE

The tonnage of the Kantienpan deposit was calculated at 5 Mt, using the geological block model (as presented in Paragraph 6.1). The average grade, calculated statistically (as presented in Paragraph 6.2), are 4.09% Zn and 0.49% Cu and a Zn equivalent value of 4.94% using the Student T method. In all the subsequent sections, the Zn equivalent will be used.

6.4 CLASSIFICATION OF THE KANTIENPAN RESOURCE

The Kantienpan resource could now be classified in terms of the South African Mineral Resource Committee (SAMREC) guidelines, using the method proposed by Dr F.A. Camisani (pers. comm., 2001), who is currently the chairman of this committee.

Dr F.A. Camisani (pers. comm. 2001) proposed the following resource and reserve classification:

Inferred Resource: $15\% \geq \bar{x} - 1.753\frac{s}{\sqrt{n}}$ for lower limit.

Indicated Resource: $10 - 15\% < \bar{x} - 1.753\frac{s}{\sqrt{n}}$ for lower limit.

Measured Resource: $10\% < \bar{x} - 1.753\frac{s}{\sqrt{n}}$ for lower limit.

Probable Reserve: $10 - 5\% < \bar{x} - 1.753\frac{s}{\sqrt{n}}$ for lower limit.

Proven Reserve: $5\% < \bar{x} - 1.753\frac{s}{\sqrt{n}}$ for lower limit.

The Kantienpan resource classification used the Student-t method, because there are so few samples. The standard deviation (s) was calculated as follows: $s = \sqrt{\frac{1}{n-1}\left(\sum_{i=1}^{k} gx_i^2 - n\bar{x}^2\right)}$ = 2.035, with n = 10 and $\bar{x}$ = 4.94. The above classification formula was then populated with the values and the calculations can be seen in the following paragraph.

From the statistics of the Kantienpan sample population the lower grade interval may be calculated at a 90% central confidence level by using the following formula: $4.94 - 1.753\frac{2.035}{\sqrt{10}}$ = 3.81

The Kantienpan deposit should be classified as an inferred resource, as the difference between the lower grade interval and the mean of the sample population exceeds 15% of the mean; (4-94-3.81)/4.94*100 = 23%.

It is suggested that further drilling is necessary to delineate the resource, especially towards the south and in depth and to increase the statistical confidence in the resource.

7 THE FINANCIAL VIABILITY OF THE KANTIENPAN DEPOSIT

7.1 INTRODUCTION

Mining projects that are financially evaluated, always use three major input factors: mine life (a factor of the resource and the production rate), total investment (as based on fixed production rates) and returns on the investment (based on the profit). These factors are, however, each estimates with a certain degree of accuracy and a single value, such as net present value, that is calculate, should be accompanied by a statement of the reliability of the estimate. This reliability can be reported in two general ways: a sensitivity analysis and a risk analysis. Both analyses were used in the evaluation of the Kantienpan deposit.

A discounted cash flow (DCF) model was first used in the financial evaluation of the deposit and the values from the DCF were then used in the sensitivity analysis and risk analysis.

Benchmarking, with worldwide Zn-Cu deposits, were done to determine mining and processing costs, which were then used in the DCF to get a realistic and accurate estimate. The way that the mining and processing costs, as well as the methods, were determined, are described in the following paragraphs.

7.1.1 Metallurgy

Metallurgy will not be discussed in great detail, owing to the fact that flotation, the suggested beneficiation procedure, has been tested and proven to produce, in the Kantienpan case, a Zn and possibly a Cu concentrate. It is suggested that a bulk sample should be taken and sent to Kumba's Rosh Pinah mine for pilot floatation test work, should the project enter a pre-feasibility study phase.

The cost for the processing was estimated at US$10.63/t, which was benchmarked against mines such as Rosh Pinah, Maranda, Black Mountain and the Clementine software from, AME Mineral Economics. Capital cost to process the ore, for a concentrate, was calculated on a process factor of 3.5 times the operational expenditure (US$17 600 000). This operational cost was again benchmarked against mines like Rosh Pinah, Maranda, and Black Mountain and found to be comparable. A plant recovery for Zn was also benchmarked against mines such as Rosh Pinah, Maranda, and Black Mountain and found to be an average of 90%, which was used in the discount cash flow (DCF) model.

7.1.1.1 Risks

Risks, as discussed in paragraph 5.2, are the flotation problems owing to possible high iron values substituting the Zn in the sphalerite and possible problems that might arise form the floatation test work. The capital cost assumption can be a factor of risk and detailed cost should be calculated for the project in the prefeasibility stage.

7.1.2 Mining

7.1.2.1 Mining Method

The mining method proposed is underhand benching, which is derived from the orebody width and the competency and strength of the surrounding wall rock. The operation can been mechanised as far as possible, using proven techniques employed at Maranda mine, where techniques such as conveyor hoisting and load-haul-dumper ("LHD") equipment have proved successful (Terblanche, 1997).

It is proposed to develop the orebody by means of a main conveyor decline, which will also act as the main intake airway. Secondary access will be provided via the main ventilation exhaust raise. The main decline will be driven parallel to the ore zone and approximately 20m from it to 260E where the orepass system will be located (Terblanche, 1997).

The main haulage will be established on 200 Level, having drawpoints at approximately 12m centres. The orebody has been divided into on-strike stopes, each 65m long, separated by 8m rib pillars with a stoping height of 120m. Sub-levels will be established at 30m centres on 160, 130, 100 and 70m levels being connected by access raises mined in the centre of the rib pillar in the ore zone. Slot raises will be driven in ore in the centre of each stope block and the underhand bench stopes will commence from these raises on 200 Level.

Provision has been made on the main haulage (200 Level) for a orepass grizzly for both ore and waste handling and for workshop facilities.

There is a small ore extension in depth to the west below 200 Level, and provision has been made for ongoing development to exploit this by means of a scooptram decline hauling ore back up to the orepass on 200 Level.

Pre-production development is estimated at two years by the end of which 3632m will have been completed, generating approximately 19 000 tons of ore, and four stope blocks will be available for mining. Ongoing development is assessed at a rate of 72m per month and ore production has been set at 6 000 tons per month.

All development and stope drilling will be carried out by jackhammers. Development cleaning and stope production will be effected by a fleet of three two-yard scooptrams (inclusive of spares for maintenance and downtime), discharging into the main orepass on 200 Level, with ore transported to surface via a 24-inch conveyor system.

The cost of mining was estimated at US$18.72/t, which was again benchmarked against mines like Rosh Pinah, Maranda, Black Mountain and the Clementine software from, AME Mineral Economics. The capital mining cost was calculated on a mining factor of 2 times the operational expenditure (US$17 700 000). This mining operational cost was again benchmarked against mines like Rosh Pinah, Maranda, and Black Mountain and found to be comparable. A mining recovery of 90%, a mining dilution of 5% and a transport cost of US$0.005/km ton were used in the DCF model. These assumptions were again benchmarked and found to be comparable against mines such as Rosh Pinah, Maranda, and Black Mountain and found to be an average of 90%, which was used in the discount cash flow model.

7.2 DISCOUNTED CASH FLOW MODEL

The DCF was based on a geostatistical derived Zn equivalent value of 5%, a tonnage of 5 Mt derived from the geological block model and a Zn flat rate price of US$950/ton. The DCF as a whole is calculated in US$ although the NPV is also given in rand. A real discount rate of 8% was used, because this is currently the hurdle rate Kumba Resources Ltd. applies for projects in the "blue skies" phase.

The tax rate of 30% was used in the calculations, which is the company tax percentage for 2002, but no tax was paid, because the project has a negative NPV. Only royalties were paid and the amount can be seen in Table 9. The transport cost factor was obtain from benchmarks with other deposits like Black Mountain and Rosh Pinah.

It is, however, important to note that the project is probably in the "blue skies" phase and the model is therefore kept very simple. The assumptions used are mostly generalised, although the values correlates with other mines that were benchmarked. The achieved results, from the model, are however more than adequate for evaluating the project at this level.

The detailed costs and assumptions of the model can be seen in Appendix G. Table 9 gives a summary of cash flow of the project and it indicates that the project will probably not be positive as a stand-alone project.

Table 9. Total Cash Flow to Equity.

Cash Flow	Amount (US$)
Discount Rate	8%
Revenue	101 841 416
Operational Expenditure	-147 789 650
Royalties	-509 207
Capital Expenditure	-39 889 000
Total	**-86 346 441**

A negative NPV of US$66 818 693 was calculated.

7.3 KANTIENPAN BENCHMARKED AGAINST WORLD ZN PRODUCERS

7.3.1 Introduction

Most of the current world Zn producers were plotted in Figure 14 against their annual production and cash cost and a trend line was then fitted to the data.

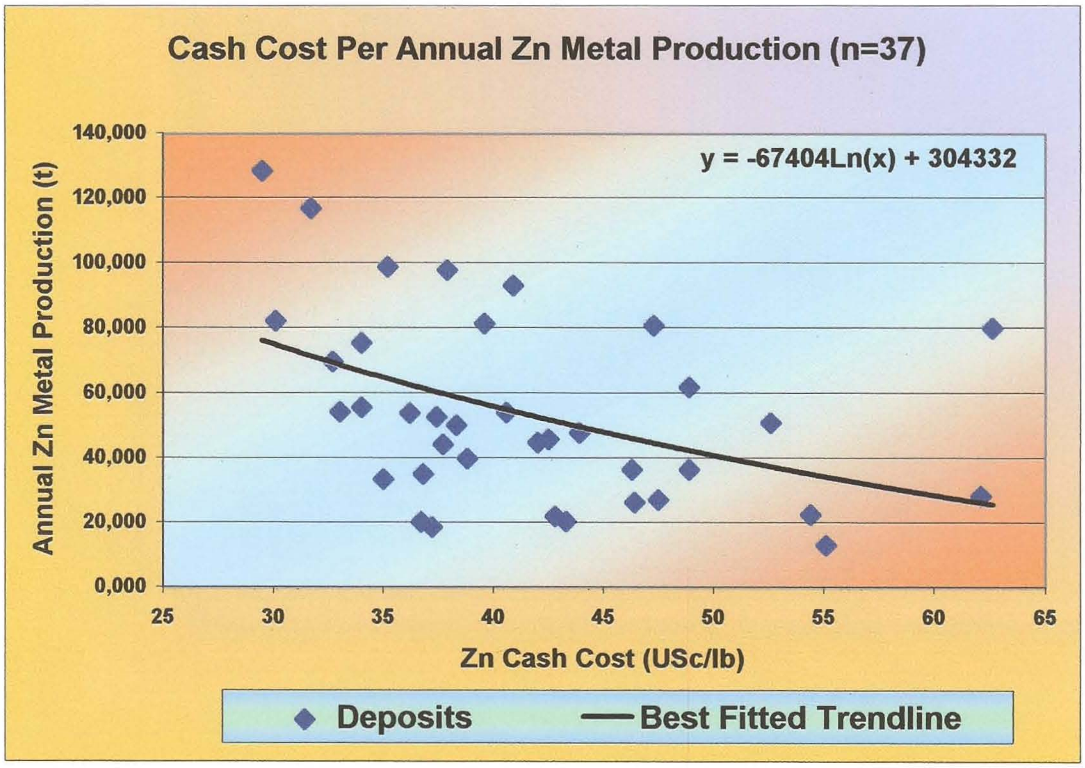

Figure 14. Cash cost per annual Zn metal production.

The life of mine (LOM) for each deposit was calculated by using the empirical formula for base metal deposits: LOM = 0.2 (Reserve)$^{0.25}$ (Noakes and Lanz, 1993). With the known LOM values, the total annual metal ton production could be calculated. The formula for the trend line was then used to calculate the cash cost for each deposit. This cash cost per pound of metal could be contoured by the solid lines in Figure 15.

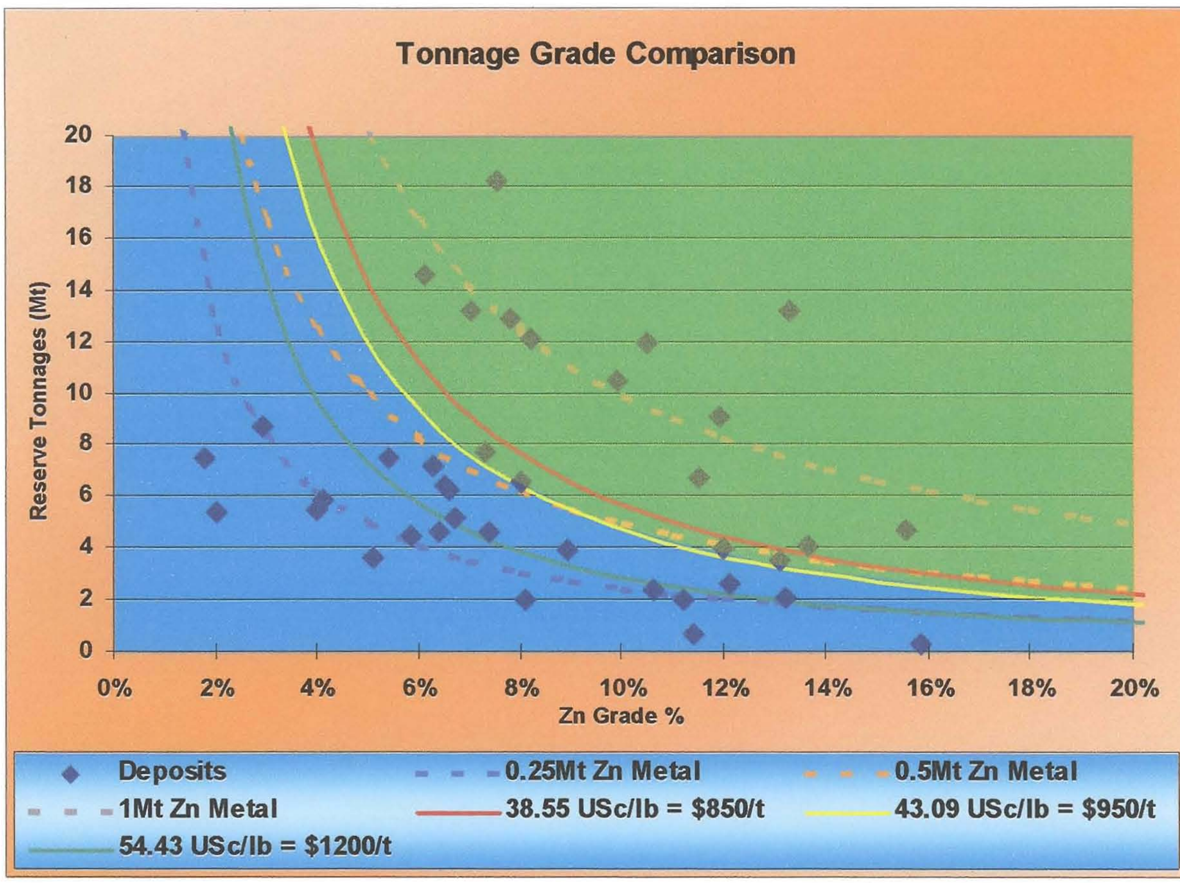

Figure 15. Tonnage and grade comparison at a Zn price of US$950/t.

All the deposits in the green area are economical when based on the Zn value, taking the Zn price to be US$950/t (Figure 15). Since all the deposits on the graph are current in production, this means that those deposits under the line must be making their money from other metals in the ore, given for instance a higher Cu price than the price for Zn. For those deposit in the green area the working cost is lower than the revenue at a Zn price of US$950/t and where the deposit plot in the blue the working cost exceed the revenue. Price selected brake even lines at are also provide for other Zn prices and comparison million ton contained Zn Metal lines were also plotted to compare different deposits.

By using Monte Carlo simulations (Appendix H), the present value contours for the Kantienpan deposit, specifically, could also be plotted (dotted lines in Figure 16).

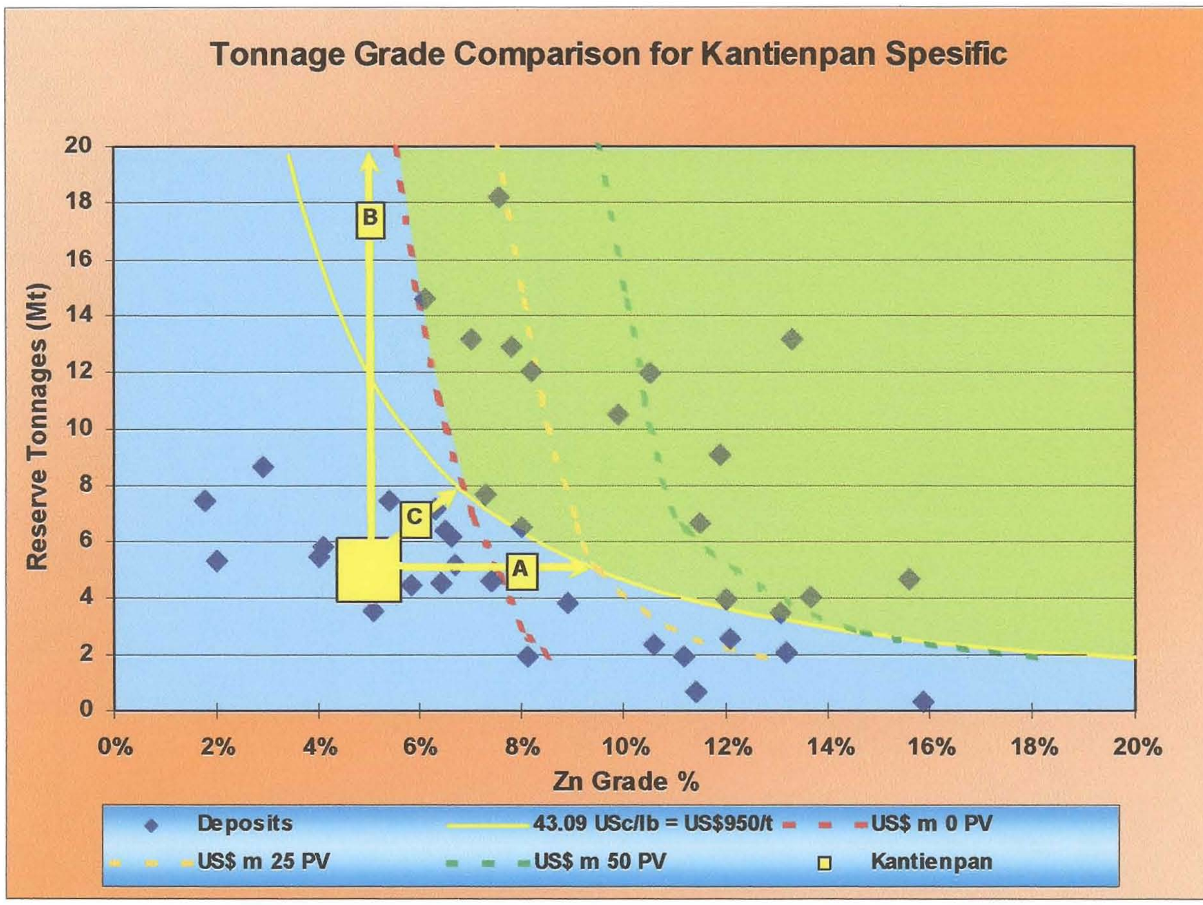

Figure 16. Tonnage and grade comparison for the Kantienpan deposit at a US$0 NPV and a Zn price of US$950/t.

The green area in Figure 16 shows the area of economic viability for the Kantienpan deposit. The yellow box is the current status of the Kantienpan deposit and the arrows, A, B and C, indicate the grades and tonnages, which will make the deposit economically feasible. It is however clear that the grade needs to increase dramatically before the deposit will be economical. Arrow C gives an indication of 7 % Zn and 8 Mt reserves to get to a breakeven situation. Another scenario is to increase the Cu grade, which will reach a positive NPV more easily than an increase in the Zn grade, due to a higher Cu price. Further exploration drilling near Cu rich areas could lead to the discovery of the fumarolic vent with higher Cu grades and a detailed drilling programme is suggested for a prefeasibility stage.

7.4 SENSITIVITY ANALYSIS

A sensitivity analysis is one way of assessing reliability, by which one variable's value is changed systematically and the corresponding value of the financial criteria is reported. These corresponding values are plotted and the steeper the gradient of the line, the more sensitive the financial criteria to the variable's value.

Sources of risk, which are the variables used in a sensitivity analysis, are costs, prices, fluctuations in exchange rates, ore reserves, mineral processing, completion time of the project, pollution abatement costs, political, etc. The sensitivities that were used in this study can be seen in Figures 17 and 18, with grade being the most sensitive variable and price the second most sensitive variable to the NPV. Ore resources are very insensitive for this specific deposit. Costs, especially operating cost, are also sensitive, but were not plotted due to the fact that costs were calculated on a mining and processing factor (see Appendix G).

The sensitivity in terms of tonnage was evaluated using realistic increments at a grade of approximately 5% Zn (increments of 5Mt, up 30Mt were considered). It was found that a relatively small increment in tonnage would not provide any significant economic benefit. Economy of scale benefits would only be notable with very large increases in the tonnage.

The prices that were used were only for Zn and not for Cu. Cu prices are higher than Zn prices, which means that the NPV will be more sensitive towards Cu prices than Zn prices.

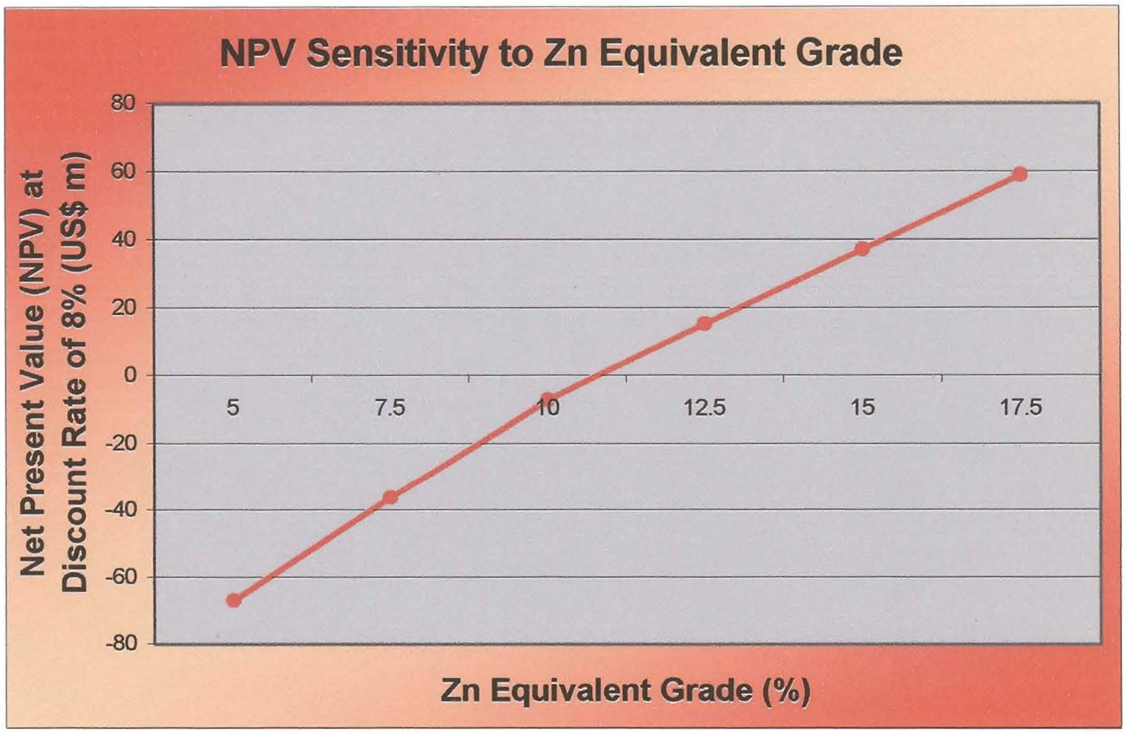

Figure 17. Zn Equivalent grade sensitivity plot.

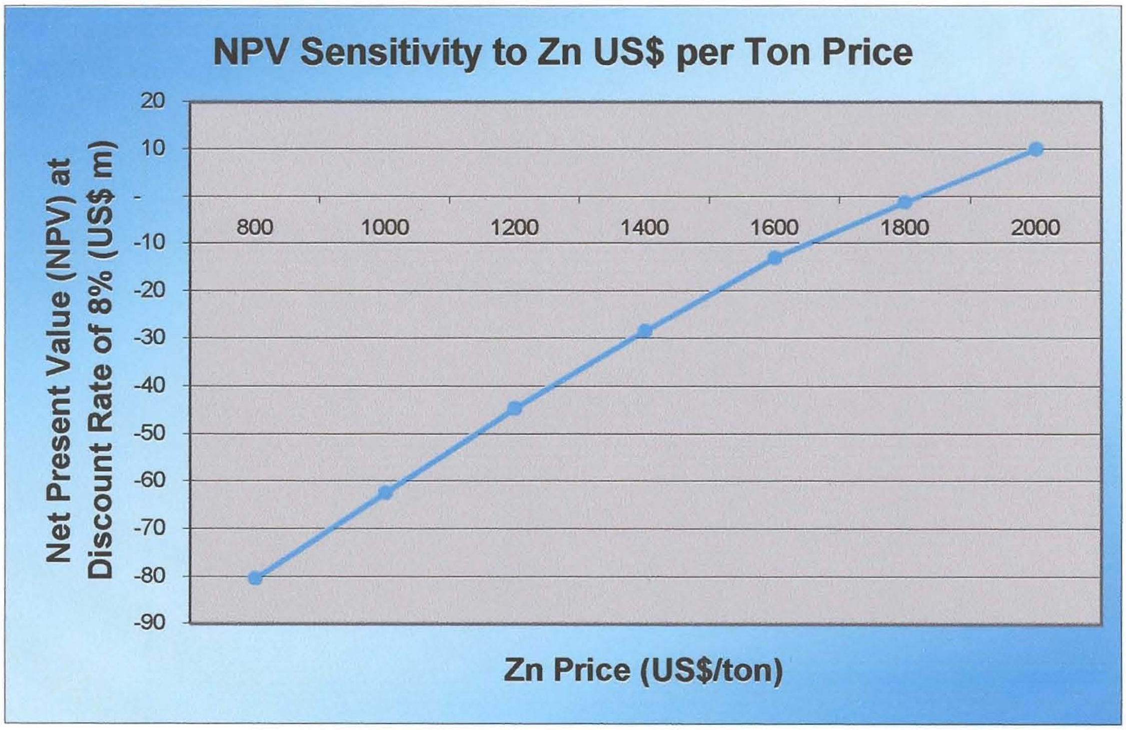

Figure 18. Zn US$/t price sensitivity plot.

7.5 RISK ANALYSIS

Risk analysis is another way of assessing the confidence in the reliability of the final recommendation. One of several mathematical techniques for performing probabilistic risk assessments is Monte-Carlo analysis. Monte-Carlo analysis allows all variables to change their values simultaneously with each variable selected at random from a histogram that summarises historical values or future values of the variable based on a prefer prediction model.

According to Vose's, the cardinal rule of risk analysis modelling is: "Every iteration of a risk analysis model must represent a scenario that could physically occur." Following of this rule will result in a risk model that is both accurate and realistic (Vose, 1996).

A Monte-Carlo analysis was used to perform a probabilistic risk assessment of the Kantienpan deposit (Appendix H). A spreadsheet was set up with the same information that was used it the DCF. A standard deviation (STD) column was added on the right-hand side of the DCF information and populated with deviations for each aspect that would simulate the model as accurate and reliable as possible. A hundred permutations were added in columns to the right of the STD column to simulate 100 scenarios. The results were summarized in a cumulative frequency and variable value range table and plotted as a probability plot (shown in Figure 19).

Two analyses were done and plotted as probability plots of the NPV value at a discount rate of 8%. The first analysis was done with the current scenario with a 5 Mt reserve and an equivalent Zn grade of 5% (Figure 19). It is clear from the results that the project only has a 2.4% change on success. These results correlates with the negative NPV value from the discounted cash flow model that the project will not be economical at a present.

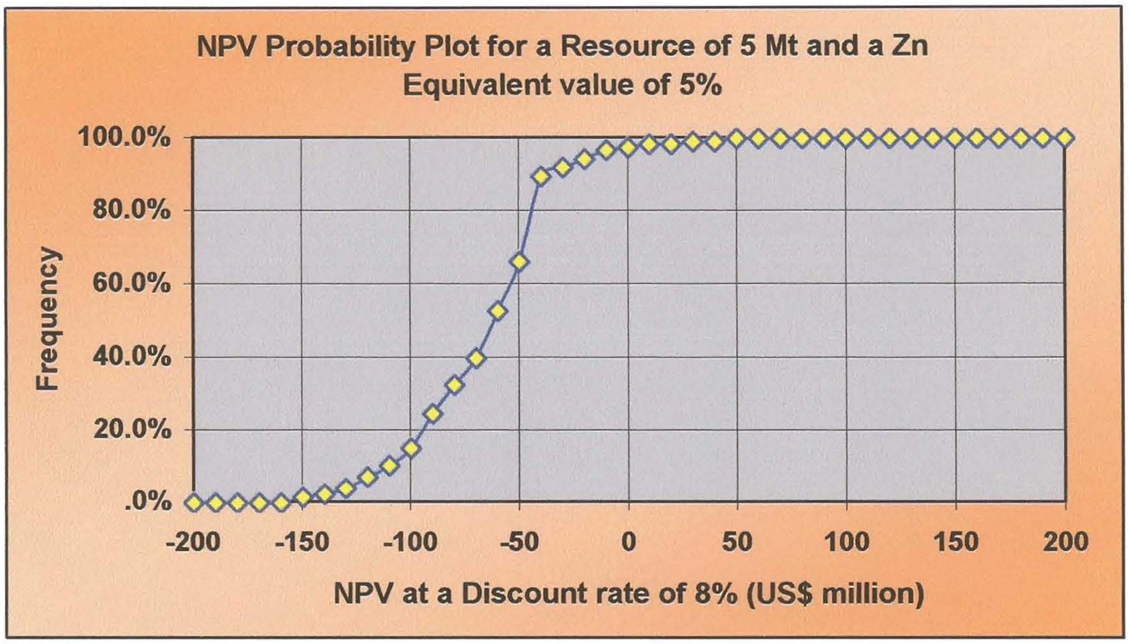

Figure 19. NPV probability plot for a reserve of 5 Mt at 5% Zn Eq.

Looking at an example of 5 Mt reserves at 12% Zn grade (Figure 20), the project will be economically viable 40% of the time. It is still less than what is needed for a project in a final feasibility stage, but most companies will jump at such a project in the "blue skies" phase.

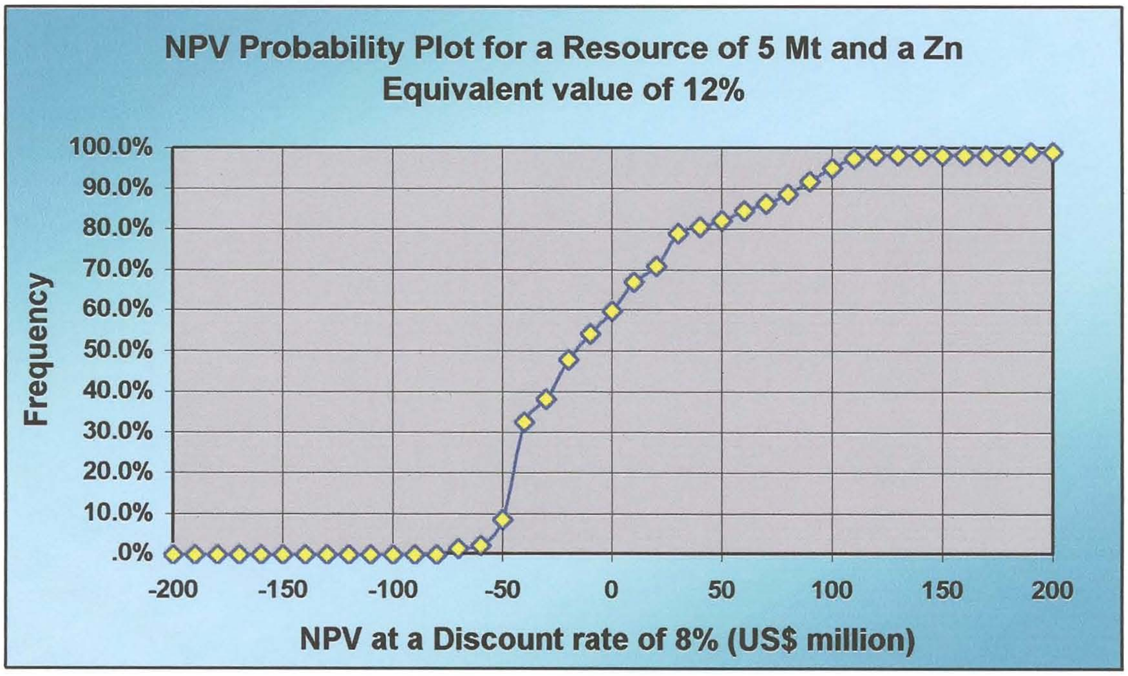

Figure 20. NPV probability plot for a reserve of 5Mt at 12% Zn Eq.

7.6 ALTERNATIVE SCENARIO

An alternative option that must also be considered is the cost of importing Zn concentrate, should Zincor run out of nearby suppliers. With the closure of Pering, imminente closure of Maranda and the decreasing life of mine of Rosh Pinah, demand for Zn concentrate, to supply the smelter, is steadily increasing.

The cost to import 1 ton of Zn concentrate cannot be reviled, but is taken at US$X/t more than for domestic concentrate (information supplied by Kumba's commodity annalist, Mr E.T. Fourie, pers. comm., 2002). The discounted cash flow model shows an operating income per ton milled of -US$9.7 (Appendix H). With a concentrate Zn grade of 55%, a Zn grade of 5% and a recovery of 90%, 12.2 run of mine tons are needed to produce one ton of concentrate (Concentrate grade / (Zn grade x Recovery)). The imported concentrate amounts to an operating cost of US$4.9/ton milled. Adding the cost to import Zn concentrate (-US$9.7 + US$4.9) gives an operation income of -US$4.8/t.

This means that a loss of US$4.8/t is still made to produce a ton even if one can save the US$X/t to import Zn concentrate. Should the Zn price increase to US$1200/t, the operating income per ton milled is -US$3.7 (Appendix H). Adding again the cost to import Zn concentrate (-US$3.7 + US$4.9) gives an operation income of US$1.2/t, which means that the project is positive. It must be stated that the Cu and Ag has not been taken into account; additional revenue from these elements would definitely affect the project positively.

8 SUMMARY

The Areachap Group and its economic potential must not be under-estimated and remains a possible source of Zn and Cu ore. The Kantienpan deposit displays convincing similarities with a VHMS deposit, chemically, structurally and lithologically. The deposit is also similar to other Zn-Cu deposits in the Areachap Group such as Prieska Zn-Cu mine and Areachap mine.

When exploring for VHMS in a highly metamorphosed and structurally complex area, like the Areachap Group, it is important to unravel the geology. The lithogeochemical method was proven to be a successful exploration tool for VHMS deposits in the Areachap Group.

The amphibolite facies metamorphism also led to annealing of the sulphide minerals, which increased the grain size and will improve the metallurgical liberation of minerals.

The financial analyses tools helped to get a good understanding of the economic and viability potential of the project. The DCF proved that the current project is not viable as a stand-alone project. The sensitivity analysis proved that the project is very sensitive to grade and metal prices and the probabilistic risk assessments showed that the project has a 2.4% chance of success at present.

The proposed alternative scenario, to mine the deposit even with a negative NPV, but at a higher commodity price, can be successful, should the Zincor smelter run out of Southern African suppliers.

9 RECOMMENDATIONS

It is recommended that the lithogeochemical database be used for further target generation on the Areachap Group as a whole.

Given the higher values of Cu, the most feasible way of reaching a positive NPV, would be to prove addition reserves at a higher Cu grade. This would require careful evaluation of the genetic model, the available geological information and the current ore body model. To prove additional resources further exploration drilling would be needed, especially in depth and towards the southern part of the deposit.

A suggested bulk sample should be taken and sent to Kumba's Rosh Pinah mine for pilot floatation test work, should the project enter a pre-feasibility stage.

The alternative scenario of mining at a loss, but supplying the smelter with cheaper concentrates, should be kept in mind.

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APPENDIX: A. MINERAL OCCURRENCES REVIEW

The summary given below is from Blignault and van Schalkwyk's report (1995): Exploration targets in the Jannelsepan amphibolite belt. They were consulted by Iscor to do a literature study of the Areachap sequence. The localities of the different deposits can be seen in Figure 4.

Deposit Name	Locality	Commodity & Deposit type	Mineralisation	Size & Grade Estimate
Areachap	Farm Areachap 426, 22 km NW of Upington	Cu-Zn massive sulphide; oceanic crust association	Conspicuous surface outcrop of gossan with sparse copper stains is the only outcrop in the area. Oxidation and leaching is developed down to 70 m with supergene copper enrichment (3,5% Cu & 2% Zn) between 70 and 90 m. Below 90 m the subvertical tabular sulphide body consists of both massive and disseminated sulphides mostly pyrite, pyrrhotite, chalcopyrite and sphalerite on average 5 m thick and surface strikes of 600 m with the deepest intersection at 750 m. The deposit is open-ended at depth.	8,9 Mt @ 0,47% Cu 2,24% Zn 4,6 g/t Ag 0,07 g/t Au.
Boksputs	Farms Boksputs 118 and Koegrabe 117	Minor Cu-Zn massive associated with extensive disseminated sulphides	The sulphide minerals are pyrite, pyrrhotite and magnetite. Massive sulphides with percentage values of Cu are developed in thin zones of 1 to 2 m, with wider areas of disseminated sulphides.	Not determined
Kantienpan	Kantienpan 119 and Gemsbok Bult 120	Cu-Zn massive sulphide	Float of gossan zones and a BIF can be seen within an amphibolite sequence of the Jannelsepan Formation.	Not determined

Deposit Name	Locality	Commodity & Deposit type	Mineralisation	Size & Grade Estimate
Kielder	Situated 12 km NW of the Prieska a/b on the farm Kielder	Zn>Cu>>Pb	Three stratabound massive sulphide bodies 2 km and 3,5 km apart; disseminated pyrite haloes	Not determined
Jacomynspan	The deposit straddles the boundary between the farms Jacomynspan 176 and Hartebeest Pan 175, west of Marydale in the Kenhardt District.	Cu-Ni sulphides in layered complex; oceanic crust association	On Jacomynspan, the main materialisation is in the form of disseminated sulphides (1 to 3%) in the biotite-tremolite schist. The hypersthene rock sporadically carries 10 to 20% total disseminated sulphides. The sulphide minerals present are pyrrhotite, chalcopyrite and pentlandite. The width of materialisation varies from 1 to 60 along a total strike length of 4 500 m.	114 Mt @ 0.25% Ni and 0.17% Cu on the Jacomynspan portion
Prieska Annex	Farm Annex Vogelstruisbult	Cu-Zn massive sulphide of oceanic crust association	Massive sulphide	1,5 Mt @ 1,5% Cu < 0,5% Zn
Witkop	Farm Eyerdoppan 58 (ptn RE)	Au-(Cu-Zn-Ag) mineralisation	Disseminated mineralisation with metamorphosed argillic alteration type. Presumably situated in the Jannelsepan Formation	One intersection resulted in 5 g/t Au over 30 m in fold nose Adjacent rock gave values of 0,69% Cu 0,10% Zn 3 g/t Ag

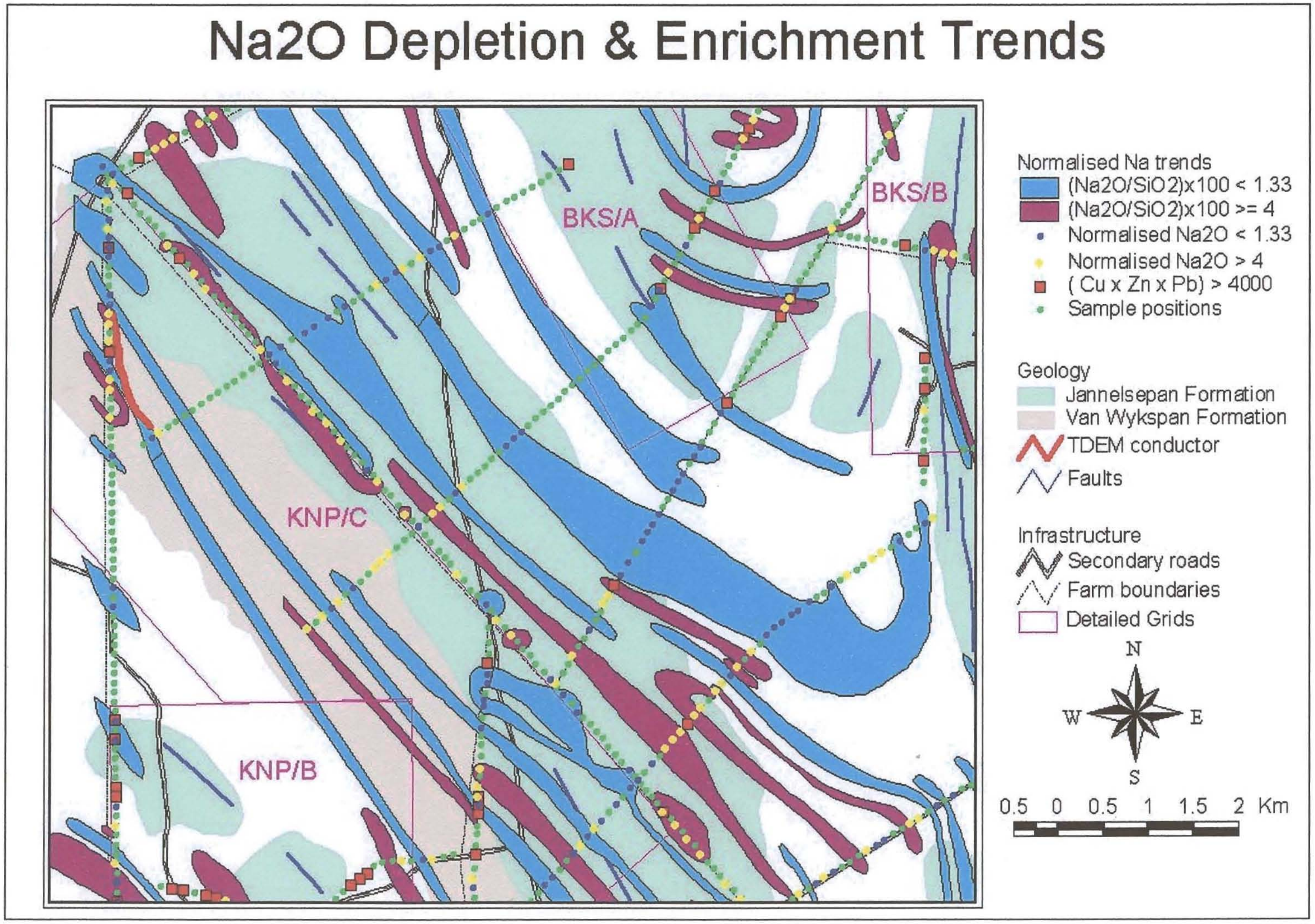
Deposit Name	Locality	Commodity & Deposit type	Mineralisation	Size & Grade Estimate
Long Anomaly Gossan Zone	Located on the farms Van Wyks Pan 170 and Rooi Puts 172	Barren massive sulphides	According to unconfirmed reports no significant intersections have yet been made	Probably a major resource of pyrite and pyrrhotite
Prieska Cu-Zn mine	Farm Vogelstruisbult	Cu-Zn massive sulphide; oceanic crust association	Small gossan showing in flat calcrete area, but well developed below calcrete; gossan siliceous with sparse malachite stains; substantially elevated levels of Cu, Pb, Zn, Ba & Mo; the Ni and Co values are flat. Leached zone for the first l00 m. Strike length of the tabular and stratabound sulphide body is 2000 m, 7 m wide on average and extends down to 1000 m. Massive sulphide (55% sulphide) zoned. Pyrite, pyrrhotite, chalcopyrite and silimanite. Some carbonate and barite in a/b. Sulphide body typically contains numerous inclusions or fragments. The so-called alteration zone is a schist with disseminated pyrite.	47 Mt @ 1,74% Cu, 3,87% Zn, 8 g/t Ag, 0,4 g/t Au & 50-100 g/t Mo.

APPENDIX B. TONNAGES AND GRADES OF SOUTH AFRICAN AND WORLD WIDE VMS DEPOSITS

(Cain, 1994)

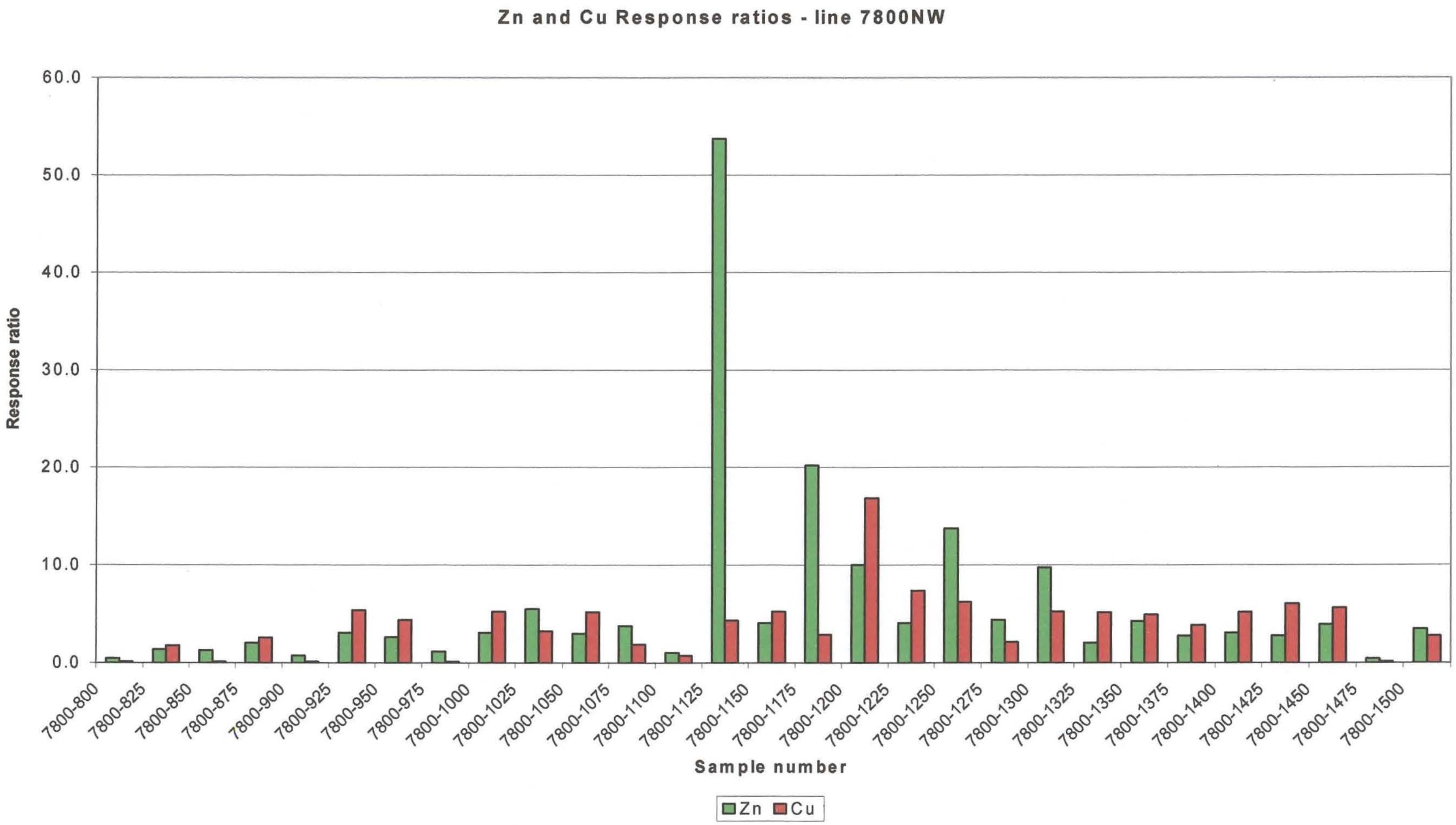
Deposit	Geological Setting	Country	Tonnage (Mt)	Zn (%)	Cu (%)	Pb (%)	g Ag/ton	g Au/ton
Average of 52	Abitibi Belt	Canada	9.20	3.43	1.47	0.07	31.7	0.31
Average of 50 (excluding Kidd Creek and Horne	Abitibi Belt	Canada	3.98	3.43	1.47	0.07	31.7	0.31
Average of 38	Norwegian Caledonides		3.46	1.53	1.41	0.05	Na	Na
Average of 29	Bathurst camp	Canada	8.70	5.43	0.56	2.17	60.03	0.47
Average of 28 (excluding Brunswick No.12)	Bathurst camp	Canada	5.72	5.43	0.56	2.17	60.03	0.47
Average of 25 major Kuroko	Green Tuff Belt	Japan	5.81	3.86	1.63	0.92	12.17	0.37
Mr. Chalmers		Australia	3.6	0.8	1.8	0.1	0.014	0.002
Maranda	Rubbervale Formation (Copper-Zinc Line)	RSA	0.5	20	3	-	25	0.2
Romotshidi	Rubbervale Formation (Copper-Zinc Line)	RSA	0.4	20	3	-	25	0.2
Prieska	Copperton Formation	RSA	47	3.87 (2.62)	1.74 (1.11)	-	8	0.4
Areachap	Areachap Group	RSA	8.9	2.24	0.4	-	4.6	0.07

APPENDIX C. TARGET GENERATION BY USING THE VMS CONCEPTUAL MODEL (Rossouw, 1999)



APPENDIX D. MMI Graph

Line 7800NW crosses the TDEM conductor at station 1125 with the highest value, proving that MMI can work to test conductors for Zn and Cu content (Rossouw, 1999).



APPENDIX E. DRILL HOLE STRATIGRAPHY OF THE KANTIENPAN VMS DEPOSIT (Rossouw, 1999)

Summary borehole log of KN005

Depth (m)	
0.00	Calcrete
6.00	Pegmatite
9.00	Pinky grey-beige, medium-grained, granular to weakly banded quartzofeldspathic gneiss with intercalated bands of feldspathic amphibolite
31.74	Dark-grey, fine-grained and weakly foliated feldspathic amphibolite. Possible sill.
42.00	Dark-grey, medium-grained, feldspar-quartz-amphibole gneiss. Displaying banding and moderate foliation. Disseminated sulphides, mainly pyrite (±2%) with subordinate pyrrhotite
58.54	Pale-grey, strongly banded feldspar-quartz-amphibole gneiss. Weakly disseminated sulphides (<1%)
71.00	Hand-axe, quartzofeldspathic gneiss marker with garnetiferous layer (74-76m)
77.23	
77.83	Dark-grey, fine-grained and weakly foliated feldspathic amphibolite. Possible sill.
81.97	Massive sulphides (80%) and wall-rock clasts (20% - chlorite and quartz). Sulphides: Pyrrhotite 45%, sphalerite 30%, pyrite 25% and chalcopyrite 5%. Sphalerite matrix to pyrrhotite and pebbly and coarse blebby pyrite replaces both. Rounded wall-rock clasts indicative of vent exhalation. Thin chert topping (5cm)
90.26	Stringer sulphides in medium-grained garnet-cordierite feldspathic amphibolite. Moderately banded and weakly foliated measly alteration. Footwall alteration.
97.00	Polymict medium to coarse-grained, moderately banded, grey gneiss with occasional coarse-grained sugary metaquartzite bands. Occasional garnet porphyroblasts. Disseminated sulphides (±1%): Pyrite and subordinate pyrrhotite
128.84	Dark-grey weakly banded moderately foliated biotite amphibolite; with some intercalated thin metaquartzites. Garnet marker 148 – 149m. Possible sill.
150.40	EOH

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APPENDIX F. PROTOLITHOLOGY AND GEOCHEMICAL INTERPRETATION OF THE AREACHAP GROUP (MODIFIED AFTER GERINGER, 1994)

	INTERPRETATION	MAIN PETROGRAPHIC FEATURES	PLOT L	ENRICHED	CENTRAL	
AMPHIBOLITE (MASSIVE)	THOLEIITIC BASALTS DERIVATION; SOME MAY BE SEDIMENTARY	QTZ-PLAGS SEGREGATIONS (AMYGDALES)	THOLEIITIC BASALT FIELD MAINLY	SLIGHT		JANNELSEPAN FORMATION
	ARC-RELATED BASALT OR REWORKED VOLCANICS OF THOLEIITIC COMPOSITION				ARC-RELATED THOLEIITE (MORB NORMAL)	
PYROXENE-AMPHIBOLITE	HIGH K BASALT OR SHOSHONITE CALC-ALKALINE AFFINITY		BORDER OF GREYWACKE / BASALT FIELD *	LARGE	HIGH K_2O	
	MARLS	INTERCALATED WITH CALC-SILICATES				
BIOTITE GNEISS (QUARRY MB) SPRIGG FORMATION	SEDIMENT FROM MAFIC SOURCE		SEDIMENTARY FIELD ***		LOW MgO, FeO AND MnO. HIGH SIO_2, K_2O. ENRICHED IN CO, V, Ni. DEPLETED IN Rb, Nb, Zr.	
	NEAR-SHELF DEPOSIT WITH CONGLOM. AS CHANNEL FILL					
QUARTZ-FELDSPAR GNEISS (SKIETBAAN MB)	RHYOLITIC TUFFS AND VOLCANICS; PARTLY REWORKED		SIMILAR TO RHYOLITE ON R1-R2****			
METAPELITE (BETHESDA FM)	REWORKED RHYOLITE / RHYODACITE	GRT-QTZ-K-FELDSPAR-PLAG-BT GN ONLY	MORB-NORMAL MINOR CORD.		VERY LOW CaO. HIGH K_2O AND Na_2O	
MASSIVE AMPHIBOLITE	LOW K THOLEIITE	"GAUBESCHIEFER" TEXTURE	ARC-RELATED THOLEIITE		MORB-NORMAL HIGHISH Zr/Ti HIGHISH SIO_2 (TO 58%)	BOKSPUTS FORMATION
PYROXENE-AMPHIBOLITE	CALC-ALKALINE BASALT	INTERCALATED WITH Fe FORMATION	SPLIT BETWEEN BASALT AND OTHER FIELDS * CALC-ALKALINE ON JENSEN PLOT **		LOWER Zr/Ti LOW SIO_2 (TO 51%)	
AMPHIBOLE GNEISS (KANTIENPAN FORMATION)	VOLCANIC-DERIVED SEDIMENTS FROM DACITE SOURCE	GRT-BEARING HBL GNEISS. HBL+BT AGGREGATES. SPECKY APPEARANCE. FRECKY GNEISSES "GARBESCHIEFER TYPE"	COMPARABLE TO DACITE. PLOTS OFF IGNEOUS TREND ON LEAKE + SINGH DIAGRAM		DEPLETED MgO, TiO_2 AND K_2O	
QUARTZ-FELDSPAR GNEISS	FELSIC VOLCANICS OF RHYODACITE COMPOSITION		RHYODACITE TO RHYOLITE ON R1-R2 DIAGRAM		NORMALIZED MAJOR ELEMENTS AS FOR RHYODACITE	
METAPELITE (KANTIENPAN FORMATION)	REWORKED RHYOLITE / RHYODACITE	GRT-CORD-BT-GNEISS WITH SILLIMANITE	SEE BETHESDA FORMATION			
MASSIVE AMPHIBOLITE	BASIC THOLEIITIC DYKES	CROSS CUTTING		FLAT PROFILES	LOW SIO_2 (<53%) HIGH CaO (6-10%) MgO (5-8%) FE_2O_3 (14-17%)	COPPERTON FORMATION
LAMINATED AMPHIBOLITE	THOLEIITIC LAVAS			FLAT PROFILES	HIGHER CaO THAN MASSIVE AMPHIBOLITE	
AMPHIBOLE GNEISS (SMOUSPAN GNEISS)	DACITE	IGNEOUS ZIRCONS	NEAR THE IGNEOUS TREND ON A LEAKE AND SINGH DIAGRAM			
QUARTZ-FELDSPAR GNEISS	HYDROTHERMALLY ALTERED ROCK NEAR OREBODY; DERIVED FROM SMOUSPAN GNEISS	STRUCTURALLY BENEATH HB-BT GNEISS (ANNEX)	BIRDWING □ PROFILE IN ANNEX -PCM RHYODACITE ON R1-R2 DIAGRAM		NORMALIZED MAJOR ELEMENTS SIMILAR TO DACITE	
METAPELITE (COPPERTON FM)	REWORKED RHYOLITE / RHYODACITE	QTZ-PLAG-BT-GRT + STAUROLITE	SEE BETHESDA FORMATION			
MICROBANDED AMPHIBOLITE	ARC THOLEIITE			FLAT PROFILES		ANNEX
HORNBLENDE-BIOTITE GNEISS (AMPHIBOLE GNEISS)	DACITIC TUFFS; PARTLY REWORKED		DACITE / RHYODACITE ON R1-R2 (DE LA ROCHE DIAGRAM)	SIMILAR PROFILES TO SMOUSPAN GNEISS		

APPENDIX G. DISCOUNTED CASH FLOW MODEL

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Assumptions

Dimension

Strike length	m	800
Width	m	300
Thickness	m	6.01
SG		3.47
Dip angle	deg	80

Geological Grades

Zn geol grade	%	4.09%
Cd geol grade	%	0.035%
Cu geol grade	%	0.49%
Pb geol grade	%	0.00%
Ag geol grade	g/t	20
Ag geol grade	oz/t	0.64
Au geol grade	g/t	0.50
Au geol grade	oz/t	0.016

Mineralogy

Zn in Sphalerite	%	59.11%
S in Sphalerite	%	33.57%
Fe in Sphalerite	%	7.00%
Mn in Sphalerite	%	0.32%
Total Sphalerite		100.00%
Cu in Chalcopyrite	%	34.50%
Fe in Chalcopyrite	%	30.50%
S in Chalcopyrite	%	35.00%
Total Chalcopyrite		100.00%
Pb in Galena	%	55.00%
S in Galena	%	45.00%
Total Galena		100.00%

Mining

Life of mine	year	10
Year of first commercial production		2
Year of last commercial production		11
Mining dilution	%	5%
Mining recovery	%	90%

Processing

Zn Recovery (in Zn concentrate)	%	90%
Cd Recovery (in Zn concentrate)	%	40%
Ag Recovery (in Zn concentrate)	%	0%
Cu Recovery (in Cu concentrate)	%	90%
Pb Recovery (in Cu concentrate)	%	0%
Ag Recovery (in Cu concentrate)	%	25%
Au Recovery (in Cu concentrate)	%	0%
Ag in tailings	%	75%

Transport of Concentrate to Smelter	km	800

Tax & Royalty

Tax Rate	%	30%
Royalty (on NSR)	%	0.50%

Conversion Factors

1 lb =	0.4536	kg
1 kg =	2.2046	lb
1 t =	2204.6	lb
1 oz =	31.1	g
1 kg =	32.15	oz
1 t =	32154	oz

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Resources & Reserves

In situ Resources		
Volume	m3	1,442,400
Tonnage	t	5,005,128
In situ Metal		
Zn	t	204,710
Cd	t	1,752
Cu	t	24,525
Pb	t	-
Ag	oz	3,218,732
Ag	kg	100,102.56
Au	oz	80,468
Au	kg	2,502.56
Recoverable Reserves		
Volume	m3	1,298,160
Tonnage	t	4,504,615
Diluted Run of Mine Reserves		
Volume	m3	1,370,280
Tonnage	t	4,754,872
Saleable Metal		
Zn	t	165,815
Cd	t	666
Cu	t	19,865
Pb	t	-
Ag	oz	724,215
Ag	kg	22,523.08
Au	oz	
Au	kg	

VHMS Project

Mining

Production Year Item	Unit	Total/Average	1 2002 FALSE	2 2003 TRUE	3 2004 TRUE	4 2005 TRUE	5 2006 TRUE	6 2007 TRUE	7 2008 TRUE	8 2009 TRUE	9 2010 TRUE	10 2011 TRUE	11 2012 TRUE
Assumptions													
Mining dilution	%	5%	0%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
Mining recovery	%	90%	0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Production Rate (ROM)													
Recoverable Reserves	tpa	4,504,615	-	450,462	450,462	450,462	450,462	450,462	450,462	450,462	450,462	450,462	450,462
Diluted Ore Produce / ROM	tpa	4,729,846	-	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985
Diluted Grades													
Diluted Zn grade	%	3.90%	0.00%	3.90%	3.90%	3.90%	3.90%	3.90%	3.90%	3.90%	3.90%	3.90%	3.90%
Diluted Cd grade	%	0.018%	0.000%	0.018%	0.018%	0.018%	0.018%	0.018%	0.018%	0.018%	0.018%	0.018%	0.018%
Diluted Cu grade	%	0.47%	0.00%	0.47%	0.47%	0.47%	0.47%	0.47%	0.47%	0.47%	0.47%	0.47%	0.47%
Diluted Pb grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Diluted Ag grade	g/t	19.05	-	19.05	19.05	19.05	19.05	19.05	19.05	19.05	19.05	19.05	19.05
Diluted Ag grade	oz/t	0.61	-	0.61	0.61	0.61	0.61	0.61	0.61	0.61	0.61	0.61	0.61
Diluted Au grade	g/t	0.48	-	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48
Diluted Au grade	oz/t	0.02	-	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02
Metal Content in Ore (ROM)													
Zn	tpa	184,239	-	18,424	18,424	18,424	18,424	18,424	18,424	18,424	18,424	18,424	18,424
Pb	tpa	-	-	-	-	-	-	-	-	-	-	-	-
Cu	tpa	22,073	-	2,207	2,207	2,207	2,207	2,207	2,207	2,207	2,207	2,207	2,207
Ag	oz/year	2,896,859	-	289,686	289,686	289,686	289,686	289,686	289,686	289,686	289,686	289,686	289,686
Au	oz/year	72,421	-	7,242	7,242	7,242	7,242	7,242	7,242	7,242	7,242	7,242	7,242
OPEX		1											
Exploration	$/t	(2.16)	-	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)	(2.16)
Development	$/t	(3.12)	-	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)	(3.12)
Drilling & Blasting	$/t	(5.52)	-	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)	(5.52)
Loading & Hauling	$/t	(5.04)	-	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)	(5.04)
Hoisting	$/t	(0.12)	-	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)	(0.12)
Maintenance	$/t	(2.76)	-	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)	(2.76)
Total	**$/t**	**(18.72)**	**-**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**	**(18.72)**
CAPEX													
Capex/industrial factor		2.00											
CAPEX	$	(17,700,000)											
Total Cost	**$/year**		**-**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**	**(8,854,271.64)**

VHMS Project

Processing (production of a Zn concentrate & a Cu concentrate)

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total/Average	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Production Rate (ROM)	tpa	4,729,846	-	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985	472,985
Recoveries in Zn Concentrate													
Zn Recovery	%	90%	0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Cd Recovery	%	40%	0%	40%	40%	40%	40%	40%	40%	40%	40%	40%	40%
Ag Recovery	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Recoveries in Cu Concentrate													
Cu Recovery	%	90%	0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Pb Recovery	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Ag Recovery	%	25%	0%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
Au Recovery	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
OPEX		1.2											
Crushing	$/t	(0.66)	-	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)
Milling	$/t	(3.30)	-	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)	(3.30)
Sulphide flotation	$/t	(5.94)	-	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)	(5.94)
Thickening	$/t	(0.66)	-	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)	(0.66)
Filtration	$/t	(0.07)	-	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)	(0.07)
Total	**$/t**	**(10.63)**	**-**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**	**(10.63)**
CAPEX													
Capex/industrial factor		3.50											
CAPEX	$	(17,600,000)											
Total Cost	**$/year**		-	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)	(5,028,772.22)

VHMS Project

Concentrate

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total/Average	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Zn Concentrate Specification													
Sphalerite in concentrate	%	90%	0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Zn	%	53.20%	0.00%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%
S	%	30.21%	0.00%	30.21%	30.21%	30.21%	30.21%	30.21%	30.21%	30.21%	30.21%	30.21%	30.21%
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-
Ag	oz/t	-	-	-	-	-	-	-	-	-	-	-	-
Cd	%	0.112%	0.000%	0.112%	0.112%	0.112%	0.112%	0.112%	0.112%	0.112%	0.112%	0.112%	0.112%
Cd	lb/t	2.47	-	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47
As	%	0.46%	0.00%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%
MgO	%	1.30%	0.00%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%
Hg	ppm	40	-	40	40	40	40	40	40	40	40	40	40
Fe	%	7.00%	0.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Mn	%	0.70%	0.00%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%
SiO_2	%	1.90%	0.00%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%
Others / Guange	%	4.716%	0.000%	4.716%	4.716%	4.716%	4.716%	4.716%	4.716%	4.716%	4.716%	4.716%	4.716%
Total %	%	100.00%	0.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%
Tonnage	tpa	31,169	-	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169
Zn Metal in Concentrate	t	165,815	-	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581
Cu Concentrate Specification													
Chalcopyrite in concentrate	%	88%	0%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%
Cu	%	30.36%	0.00%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%
S in Chalcopyrite	%	30.80%	0.00%	30.80%	30.80%	30.80%	30.80%	30.80%	30.80%	30.80%	30.80%	30.80%	30.80%
Fe in Chalcopyrite	%	26.84%	0.00%	26.84%	26.84%	26.84%	26.84%	26.84%	26.84%	26.84%	26.84%	26.84%	26.84%
Au	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au	oz/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag	g/t	344.22	0.00	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22
Ag	oz/t	11.07	0.00	11.07	11.07	11.07	11.07	11.07	11.07	11.07	11.07	11.07	11.07
Pb	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Zn	%	0.99%	0.00%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%
As	%	0.03%	0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
S in Pb, Zn & As sulphides	%	0.58%	0.00%	0.58%	0.58%	0.58%	0.58%	0.58%	0.58%	0.58%	0.58%	0.58%	0.58%
Fe in Zn & As sulphides	%	1.00%	0.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%
Hg	ppm	10	-	10	10	10	10	10	10	10	10	10	10
Bi	%	0.02%	0.00%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%
Sb	%	0.03%	0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
Cl	ppm	8	-	8	8	8	8	8	8	8	8	8	8
F	ppm	10	-	10	10	10	10	10	10	10	10	10	10
Co + Ni	%	0.03%	0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
Others / Guange	%	5.60%	0.00%	5.60%	5.60%	5.60%	5.60%	5.60%	5.60%	5.60%	5.60%	5.60%	5.60%
Total %	%	100.00%	0.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%
Tonnage	tpa	6,543	-	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543
Cu Metal in Concentrate	t	19,865	-	1,987	1,987	1,987	1,987	1,987	1,987	1,987	1,987	1,987	1,987
Pb Metal in Concentrate	t	-	-	-	-	-	-	-	-	-	-	-	-
Ag Metal in Concentrate	oz	724,215	-	72,421	72,421	72,421	72,421	72,421	72,421	72,421	72,421	72,421	72,421
Au Metal in Concentrate	oz	-	-	-	-	-	-	-	-	-	-	-	-

VHMS Project

Metal Prices

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total/Average	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Metal Price													
0 Zn EF Feb 2002	$/t		1,007	1,121	1,088	1,060	1,125	1,211	1,376	1,531	1,417	1,368	1,172
0 Zn EF Feb 2002 * 0.9	$/t		906	1,009	979	954	1,013	1,090	1,238	1,378	1,275	1,231	1,055
0 Zn EF June 2001	$/t		1,000	1,000	1,075	1,045	1,063	1,137	1,259	1,415	1,435	1,213	1,213
1 Zn flat price	$/t	950	950	950	950	950	950	950	950	950	950	950	950
Zn price	**$/t**	**950**	**950**	**950**	**950**	**950**	**950**	**950**	**950**	**950**	**950**	**950**	**950**
0 Cu EF Feb 2002	$/t		1,414	1,495	1,855	1,965	1,912	1,922	1,972	2,278	2,497	2,329	2,175
0 Cu EF Feb 2002 * 0.9	$/t		1,273	1,346	1,670	1,769	1,721	1,730	1,775	2,050	2,247	2,096	1,958
1 Cu flat price	$/t	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400
Cu price	**$/t**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**	**1,400**
0 Pb EF Feb 2002	$/t		535	591	666	707	670	596	511	441	420	586	658
0 Pb EF Feb 2002 * 0.9	$/t		482	532	599	636	603	536	460	397	378	527	592
1 Pb flat price	$/t	460	460	460	460	460	460	460	460	460	460	460	460
Pb price	**$/t**	**460**	**460**	**460**	**460**	**460**	**460**	**460**	**460**	**460**	**460**	**460**	**460**
Cd price	**$/lb**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**	**0.20**
Ag price	**$/oz**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**	**4.65**
Au price	**$/oz**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**	**300.00**

VHMS Project

Zinc TC & RC

Electrolytic Process (Roast - Acid Leach - Electrowinning from Sulphide Solution)
Only high grade concentrates (Zn>50%) with combined Cu+Pb<3% are acceptable.
Hystorically, Net Smelter Return has usually been 40-60% of the value of the total zinc metal contained in the concentrate.

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total/Average	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Metal Price													
Zn	$/t		-	950	950	950	950	950	950	950	950	950	950
Ag	$/oz		-	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65
Cd	$/lb		-	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Concentrate Specification													
Zn	%		0.00%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%	53.20%
Ag	g/t		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag	oz/t		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Cd	%		0.00%	0.11%	0.11%	0.11%	0.11%	0.11%	0.11%	0.11%	0.11%	0.11%	0.11%
Cd	lb/t		0.00	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47	2.47
As	%		0.00%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%	0.46%
MgO	%		0.00%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%	1.30%
Hg	ppm		-	40	40	40	40	40	40	40	40	40	40
Fe	%		0.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Mn	%		0.00%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%	0.70%
SiO_2	%		0.00%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%	1.90%
Metal Content	t		-	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581	16,581
Net Smelter Return													
Payable Metal													
Zn paid test	%		-8.00%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%
Zn paid	%		0.00%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%	45.20%
Zn paid	$/t		-	429.39	429.39	429.39	429.39	429.39	429.39	429.39	429.39	429.39	429.39
Ag paid test	oz		-	-	-	-	-	-	-	-	-	-	-
Ag paid	oz		-	-	-	-	-	-	-	-	-	-	-
Ag paid	$/t		-	-	-	-	-	-	-	-	-	-	-
Cd paid	lb/t		-	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73
Cd paid	$/t		-	0.35	0.35	0.35	0.35	0.35	0.35	0.35	0.35	0.35	0.35
Total paid	**$/t**		**-**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**	**429.74**
Treatment Charge	**$/t**	**(159.00)**	**-**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**	**(159.00)**
Price Participation													
	$/t		87.50	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00
	$/t		**-**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**	**21.00**
RC													
Ag	$/oz	(0.30)	-	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)	(0.30)
Cd	$/lb	(2.50)	-	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)	(2.50)
Ag	$/t		-	-	-	-	-	-	-	-	-	-	-
Cd	$/t		-	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)	(4.31)
Total	**$/t**		**-**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**	**(4.31)**
Penalties													
As	$/t		-	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)	(3.90)
MgO	$/t		-	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)	(15.00)
Hg	$/t		-	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)	(2.00)
Fe	$/t		-	-	-	-	-	-	-	-	-	-	-
Mn	$/t		-	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)	(3.00)
SiO_2	$/t		-	-	-	-	-	-	-	-	-	-	-
Total	**$/t**		**-**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**	**(23.90)**
Net Smelter Return	**$/t concentrate**		**0.00**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**	**263.52**
Ratio NSR / Value ton Concentrate			**0.00%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**	**52.14%**
Zn Metal Paid	**t**		**-**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**	**8,646**
Zn Metal Paid	**lb**		**-**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**	**19,060,744**

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VHMS Project

Copper TC & RC

Smelting & Electrowinning

Most copper concentrates have copper content ranging between 28% - 34%, maximum Pb accepted is 6%
Historically, the Net Smelter Return has been 70% - 80% of the value of the metal contained in the concentrate

Production Year Item	Unit	Total/Average	1 2002 FALSE	2 2003 TRUE	3 2004 TRUE	4 2005 TRUE	5 2006 TRUE	6 2007 TRUE	7 2008 TRUE	8 2009 TRUE	9 2010 TRUE	10 2011 TRUE	11 2012 TRUE
Metal Price													
Cu	$/t		-	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400
Cu	$/lb		-	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64
Au	$/oz		-	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00
Au	$/g		-	9.65	9.65	9.65	9.65	9.65	9.65	9.65	9.65	9.65	9.65
Ag	$/oz		-	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65	4.65
Ag	$/g		-	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15
Concentrate Specification													
Cu	%		0.00%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%	30.36%
Au	g/t		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag	g/t		0.00	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22	344.22
Pb	%		0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Zn	%		0.00%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%	0.99%
As	%		0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
Hg	ppm		0	10	10	10	10	10	10	10	10	10	10
Bi	%		0.00%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%	0.02%
Sb	%		0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
Cl	ppm		0	8	8	8	8	8	8	8	8	8	8
F	ppm		0	10	10	10	10	10	10	10	10	10	10
Co + Ni	%		0.00%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%	0.03%
Net Smelter Return													
Payable Metal													
Cu paid test	%		-1.10%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%
Cu paid	%		0.00%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%	29.26%
Cu	$/t		0.00	409.64	409.64	409.64	409.64	409.64	409.64	409.64	409.64	409.64	409.64
Au paid	g/t		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au	$/t		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag paid test	g/t		-30.00	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80
Ag paid	g/t		0.00	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80	309.80
Ag	$/t		-	46.32	46.32	46.32	46.32	46.32	46.32	46.32	46.32	46.32	46.32
Total Paid	**$/t**		**0.00**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**	**455.96**
Treatment & Refining Charge													
	$/lb Cu	(0.25)	-	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)	(0.25)
	$/t Cu	(551.15)	-	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)	(551.15)
	$/t	-	-	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**	**(161.27)**
Price Participation													
if price > $ 0.9/lb	$/lb		-	-	-	-	-	-	-	-	-	-	-
if price >=$ 0.8/lb but <$ 0.9/lb	$/lb		-	-	-	-	-	-	-	-	-	-	-
if price <$ 0.8/lb	$/lb		0.80	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16
PP/lb Cu	**$/lb**		-	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**	**0.16**
PP/t concentrate	**$/t**		-	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**	**10.64**
Refining Charge for Precious Metals													
Au	$/t	(5.00)	-	-	-	-	-	-	-	-	-	-	-
Ag	$/t	(0.30)	-	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)	(2.99)
Total	**$/t**		-	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**	**(2.99)**
Penalties													
Zn	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Pb	$/t	-	-	-	-	-	-	-	-	-	-	-	-
As	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Sb	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Bi	$/t	-	-	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)	(0.20)
F	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Hg	$/t	-	-	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)	(1.00)
Cl	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Co + Ni	$/t	-	-	-	-	-	-	-	-	-	-	-	-
Total	**$/t**	-	-	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**	**(1.20)**
Total Deductions	**$/t**	-	-	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**	**(154.81)**
Net Smelter Return	**$/t concentrate**	-	-	301.15	301.15	301.15	301.15	301.1	301.1	301.1	301.1	301.1	301.1
Ratio NSR / Value ton Concentrate		0.00%	0.00%	70.85%	70.85%	70.85%	70.85%	70.85%	70.85%	70.85%	70.85%	70.85%	70.85%

VHMS Project

Transport

Production Year / Item	Unit	Total/Average	1	2	3	4	5	6	7	8	9	10	11
			2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Zn Concentrate													
Tonnage	tpa	311,688	-	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169	31,169
Distance to Smelter	km	800	-	800	800	800	800	800	800	800	800	800	800
Transport Cost	$/km t	(0.005)	-	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)	(0.005)
Cost	**$/year**	**(1,246,752)**	**-**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**	**(124,675)**
Cu Concentrate													
Tonnage	tpa	65,433	-	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543	6,543
Distance to Smelter	km	800	-	800	800	800	800	800	800	800	800	800	800
Transport Cost	$/km t	(0.005)	-	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)	(0.01)
Cost	**$/year**	**(261,731)**	**-**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**	**(26,173)**
Total Transport Cost	**$/year**	**(1,508,483)**	**-**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**	**(150,848)**

VHMS Project

Tax

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Revenue	$	101,841,416	-	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142
CAPEX for capital redemption	$	(29,652,000)											
CAPEX straigth-line depreciation	$	(10,237,000)											
Total CAPEX	$	**(39,889,000)**											
Deductions													
OPEX	$	(147,789,650)	-	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)
Royalty	$	(509,207)	-	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)
Depreciation (straight line)	$	(10,237,000)	-	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)	(1,023,700)
Capital Redemption	$	(29,652,000)	(29,652,000)	-	-	-	-	-	-	-	-	-	-
Losses Carried Forward	$	(1,328,762,952)	-	(29,652,000)	(35,321,444)	(40,990,888)	(46,660,332)	(52,329,776)	(57,999,220)	(63,668,665)	(69,338,109)	(75,007,553)	(80,676,997)
Taxable Income	$		**(29,652,000)**	**(35,321,444)**	**(40,990,888)**	**(46,660,332)**	**(52,329,776)**	**(57,999,220)**	**(63,668,665)**	**(69,338,109)**	**(75,007,553)**	**(80,676,997)**	**(86,346,441)**
Tax Paid	$		-	-	-	-	-	-	-	-	-	-	-

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VHMS Project

All Equity Cash Flow

Production Year			1	2	3	4	5	6	7	8	9	10	11
Item	Unit	Total	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
			FALSE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Metal Revenue													
Zinc Revenue	$	82,136,559	-	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656	8,213,656
Copper Revenue	$	19,704,857	-	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486	1,970,486
Total	**$**	**101,841,416**	**-**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**	**10,184,142**
OPEX													
Mining	$	(88,542,716)	-	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)	(8,854,272)
Processing	$	(50,287,722)	-	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)	(5,028,772)
Rehabilitation	$	(6,941,522)	-	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)	(694,152)
Transport	$	(1,508,483)	-	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)	(150,848)
Marketing	$	(509,207)	-	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)
Total	**$**	**(147,789,650)**	**-**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**	**(14,778,965)**
	$/lb Zn	(0.7754)	-	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)	(0.7754)
Tax & Royalty													
Royalties	$	(509,207)	-	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)
Tax paid	$	-	-	-	-	-	-	-	-	-	-	-	-
Total	**$**	**(509,207)**	**-**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**	**(50,921)**
CAPEX													
Mining	$	(17,700,000)	(17,700,000)	-	-	-	-	-	-	-	-	-	-
Processing	$	(17,600,000)	(17,600,000)	-	-	-	-	-	-	-	-	-	-
Mine Infrastructure	$	(2,824,000)	(2,824,000)	-	-	-	-	-	-	-	-	-	-
Rehabilitation	$	(1,765,000)	(1,765,000)	-	-	-	-	-	-	-	-	-	-
Working Capital	$		-	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)	(2,216,845)
Working Capital Increment	$	-	-	(2,216,845)	-	-	-	-	-	-	-	-	-
Total	**$**	**(39,889,000)**	**(39,889,000)**	**(2,216,845)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
$/ROM annum	$/t	(8.43)											
Total Cash Flow to Equity													
Revenue	$	101,841,416	-	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142	10,184,142
OPEX	$	(147,789,650)	-	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)	(14,778,965)
Tax & Royalty	$	(509,207)	-	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)	(50,921)
CAPEX	$	(39,889,000)	(39,889,000)	(2,216,845)	-	-	-	-	-	-	-	-	-
Total	**$**	**(86,346,441)**	**(39,889,000)**	**(6,862,589)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**	**(4,645,744)**
Income	$/t	(18.26)											

NPV @	5.00%	-$72,930,802
		R -729,308,018
NPV @	8.00%	-$66,818,693
		R -668,186,929
NPV @	12.00%	-$60,250,470
		R -602,504,697
NPV @	15.00%	-$56,222,704
		R -562,227,044
NPV @	#DIV/0!	#DIV/0!
		#DIV/0!
IRR	%	#DIV/0!

APPENDIX H. MONTE CARLO RISK ANALYSIS

UNIVERSITEIT VAN PRETORIA
UNIVERSITY OF PRETORIA
YUNIBESITHI YA PRETORIA

Calculated for a Zn Price of US$950/t

Cashflow Tool

	DATA	Unit	STD	Scenario's 1 - 100	
Mineral Resource	5,005,128	tons	5.00E+06	10,420,140	7,040,463
Insitu Grade					
Zn (%)	5	%	2	5.30	1.36
Pb (%)	0.0001	%	0	0.00	0.00
Ag (g/t)	0.0001	%	0	0.00	0.00
Au (g/t)		%	0	0.00	0.00
Cu (%)	0	%	0	0.00	0.00
Ore Recovery	90	%	1	89.06	90.97
Dilution at specified recovery	5	%	1	5.71	4.31
ROM Reserve	4,504,615	tons		9,874,363	6,707,906
ROM-Plant Feed Grade					
Zn (%)	4.76			5.01	1.30
Pb (%)	0.00			0.00	0.00
Ag (g/t)	0.00			0.00	0.00
Au (g/t)	0.00			0.00	0.00
Cu (%)	0.00			0.00	0.00
ROM Production	468,480	tons		468480.00	468480.00
Planned	468,480		0	468480.00	468480.00
Additional	0.00			0.00	0.00
Tons Milled	468,480	tons		468480.00	468480.00
From ROM Production	468,480			468480.00	468480.00
Additional	0.00			0.00	0.00
LOM (years) - Rom Reserves	9.62	years		21.08	14.32
LOM (years) - Milled Tons	9.62	years		21.08	14.32
Plant Variables					
Zn Recovery	90	%	1	90.53	90.81
Zn Concentrate Grade	54	%	0.5	53.39	53.35
Pb Recovery	0.1	%	0	0.10	0.10
Pb Concentrate Grade	0.1	%	0	0.10	0.10
Ag Recovery (to PbC)	40	%	0	40.00	40.00
Au Recovery (to PbC)	0.001	%	0	0.00	0.00
Cu recovery (to PbC)	0.001	%	0	0.00	0.00
Zn Concentrate Production					
Dry tons Zn Concentrate	37180.95	tons		39809.15	10387.37
Zn grade	54.00			53.39	53.35
Mositure	5.00	%	2	3.28	6.34
Wet tons Zn Concentrate	39040.00	tons		41114.91	11046.19
Pb Concentrate Production					
Dry tons Pb Concentrate	0.45	tons		0.44	0.45
Pb grade	0.10	%		0.10	0.10
Ag in Pb concentrate	40.00	g/t		40.00	40.00
Au in Pb concentrate	0.00	g/t		0.00	0.00
Cu in Pb concentrate	0.00	%		0.00	0.00
Moisture	8.00	%	0	8.00	8.00
Wet tons Pb Concentrate	0.48	tons		0.48	0.49

Metal Prices					
Zn Price (US $/t)	950		100	834.18	1150.71
Pb Price (US $/t)	0		0	0.00	0.00
Ag Price (US $/oz)	0		0	0.00	0.00
Au Price (US $/oz)	0		0	0.00	0.00
Cu Price (US$ ton)	0		0	0.00	0.00
Exchange Rate	10		2	10.01	10.24
On Mine Costs					
Total Mining Cost	87699456			87699456	87699456
Total Mining Cost/ ton ROM	187.2			187.2	187.2
Development and Silling	0		0	0.00	0.00
Longhole Drilling -Simba	0		0	0.00	0.00
DTH Drilling	0		0	0.00	0.00
Benching Costs - including Simba	0			0	0
Mining Cost -Other	87699456			87699456	87699456
Total Milling Cost	49799424			49799424	49799424
Total Maintanance Cost	0			0	0
Total Administrative Cost	0			0	0
Total On Mine Cost	137,498,880			137498880	137498880
Total On Mine Cost/ton milled	293.5			293.5	293.5
Off-mine Costs					
Transport of Zn Concentrate (R)	1952000			2276450	1971629
Transport of Pb Concentrate (R)	0			0	0
TC for Zn Concentrate (Rand)	68124800			79530545	70469635
TC for Pb Concentrate (Rand)	0			0	0
TC for Ag (Rand)	0			0	0
TC for Au (Rand)	0			0	0
TC for Cu (Rand)	0			0	0
Total Off-mine Cost	70,076,800			81806996	72441264
Total Off-mine Cost/ton milled	150			175	155
Smelter Returns					
Zn Revenue (Rand)	162127543			174928777	192424151
Zn Revenue (R/ton milled)	346			373	411
Pb Revenue (Rand)	0			0	0
Pb Revenue (Rand/ton)	0			0	0
Ag Revenue (Rand)	0			0	0
Ag Revenue (Rand/ton)	0			0	0
Au Revenue (Rand)	0			0	0
Au Revenue (Rand/ton)	0			0	0
Cu Revenue (Rand)	0			0	0
Cu Revenue (Rand/ton)	0			0	0
Total Smelter Returns Rand	162,127,543			174928777	192424151
Total Smelter Returns R/ton milled	346			373	411
Turnover	16.21	$ m		174.93	192.42
Total Revenue/ton milled	35	$/t		373	411
Cost of Sales	21	$ m		219	210
Total Cost/ton milled	44	$/t		468	448
Operating Income EBIT	-4.54	$ m		-44.38	-17.52
Operating Income/ton Milled EBIT/ton mil	-9.70	$/t		-95	-37

UNIVERSITEIT VAN PRETORIA
UNIVERSITY OF PRETORIA
YUNIBESITHI YA PRETORIA

Calculated for a Zn Price of US$1100/t

Cashflow Tool

	DATA	Unit	STD	Scenario's 1 - 100	
Mineral Resource	5,005,128	tons	5.00E+06	5,254,010	5,796,757
Insitu Grade					
Zn (%)	5	%	2	3.12	5.52
Pb (%)	0.0001	%	0	0.00	0.00
Ag (g/t)	0.0001	%	0	0.00	0.00
Au (g/t)		%	0	0.00	0.00
Cu (%)	0	%	0	0.00	0.00
Ore Recovery	90	%	1	89.86	89.57
Dilution at specified recovery	5	%	1	4.44	3.72

ROM Reserve	4,504,615	tons		4,954,527	5,407,429
ROM-Plant Feed Grade					
Zn (%)	4.76			2.99	5.32
Pb (%)	0.00			0.00	0.00
Ag (g/t)	0.00			0.00	0.00
Au (g/t)	0.00			0.00	0.00
Cu (%)	0.00			0.00	0.00
ROM Production	468,480	tons		468480.00	468480.00
Planned	468,480		0	468480.00	468480.00
Additional	0.00			0.00	0.00
Tons Milled	468,480	tons		468480.00	468480.00
From ROM Production	468,480			468480.00	468480.00
Additional	0.00			0.00	0.00
LOM (years) - Rom Reserves	9.62	years		10.58	11.54
LOM (years) - Milled Tons	9.62	years		10.58	11.54

Plant Variables					
Zn Recovery	90	%	1	89.70	88.47
Zn Concentrate Grade	54	%	0.5	53.65	53.02
Pb Recovery	0.1	%	0	0.10	0.10
Pb Concentrate Grade	0.1	%	0	0.10	0.10
Ag Recovery (to PbC)	40	%	0	40.00	40.00
Au Recovery (to PbC)	0.001	%	0	0.00	0.00
Cu recovery (to PbC)	0.001	%	0	0.00	0.00
Zn Concentrate Production					
Dry tons Zn Concentrate	37180.95	tons		23405.99	41599.10
Zn grade	54.00			53.65	53.02
Mositure	5.00	%	2	5.62	2.93
Wet tons Zn Concentrate	39040.00	tons		24721.74	42816.26
Pb Concentrate Production					
Dry tons Pb Concentrate	0.45	tons		0.45	0.45
Pb grade	0.10	%		0.10	0.10
Ag in Pb concentrate	40.00	g/t		40.00	40.00
Au in Pb concentrate	0.00	g/t		0.00	0.00
Cu in Pb concentrate	0.00	%		0.00	0.00
Moisture	8.00	%	0	8.00	8.00
Wet tons Pb Concentrate	0.48	tons		0.48	0.49

Metal Prices					
Zn Price (US $/t)	1100		100	1179.16	1063.78
Pb Price (US $/t)	0		0	0.00	0.00
Ag Price (US $/oz)	0		0	0.00	0.00
Au Price (US $/oz)	0		0	0.00	0.00
Cu Price (US$ ton)	0		0	0.00	0.00
Exchange Rate	10		2	11.33	8.15
On Mine Costs					
Total Mining Cost	87699456			87699456	87699456
Total Mining Cost/ ton ROM	187.2			187.2	187.2
Development and Silling	0		0	0.00	0.00
Longhole Drilling -Simba	0		0	0.00	0.00
DTH Drilling	0		0	0.00	0.00
Benching Costs - including Simba	0			0	0
Mining Cost -Other	87699456			87699456	87699456
Total Milling Cost	49799424			49799424	49799424
Total Maintanance Cost	0			0	0
Total Administrative Cost	0			0	0
Total On Mine Cost	137,498,880			137498880	137498880
Total On Mine Cost/ton milled	293.5			293.5	293.5
Off-mine Costs					
Transport of Zn Concentrate (R)	1952000			1583463	2277551
Transport of Pb Concentrate (R)	0			0	0
TC for Zn Concentrate (Rand)	76908800			70655340	73123363
TC for Pb Concentrate (Rand)	0			0	0
TC for Ag (Rand)	0			0	0
TC for Au (Rand)	0			0	0
TC for Cu (Rand)	0			0	0
Total Off-mine Cost	78,860,800			72238803	75400914
Total Off-mine Cost/ton milled	168			154	161
Smelter Returns					
Zn Revenue (Rand)	187726629			180952214	168891157
Zn Revenue (R/ton milled)	401			386	361
Pb Revenue (Rand)	0			0	0
Pb Revenue (Rand/ton)	0			0	0
Ag Revenue (Rand)	0			0	0
Ag Revenue (Rand/ton)	0			0	0
Au Revenue (Rand)	0			0	0
Au Revenue (Rand/ton)	0			0	0
Cu Revenue (Rand)	0			0	0
Cu Revenue (Rand/ton)	0			0	0
Total Smelter Returns Rand	187,726,629			180952214	168891157
Total Smelter Returns R/ton milled	401			386	361
Turnover	18.77	$ m		180.95	168.89
Total Revenue/ton milled	40	$/t		386	361
Cost of Sales	22	$ m		210	213
Total Cost/ton milled	46	$/t		448	454
Operating Income EBIT	-2.86	$ m		-28.79	-44.01
Operating Income/ton Milled EBIT/ton mil	-6.11	$/t		-61	-94