Mineral Engineering Technical Services (METS)
A division of the Midas Engineering Group

Haib Copper Project 2018 Preliminary Economic Assessment

For

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Prepared by

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IMPORTANT NOTICE

This notice is an integral component of the Haib Copper Project 2018 Preliminary Economic Assessment and should be read in its entirety and must accompany every copy made of the Haib Copper Project 2018 Preliminary Economic Assessment. The Haib Copper Project 2018 Preliminary Economic Assessment has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Haib Copper Project 2018 Preliminary Economic Assessment has been prepared for Deep-South Resources Inc. (Deep-South) by Mineral Engineering Technical Services (METS); and P&E Walker Consultancy and Obsidian Consulting Services as the Report Contributors. The Haib Copper Project 2018 Preliminary Economic Assessment is based on information and data supplied to the Report Contributors by Deep-South and other parties. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the services of the Report Contributors, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. Each portion of the report is intended for use by Deep-South subject to the terms and conditions of its contracts with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the report, by any third party, is at that party's sole risk.

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The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Haib Copper Project 2018 Preliminary Economic Assessment. Where the Report Contributors have made those estimates and approximations, they are subject to qualifications and assumptions and it should also be noted that all estimates and approximations contained in the Haib Copper Project 2018 Preliminary Economic Assessment will be prone to fluctuations with time and changing industry circumstances.

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The Haib Copper Project 2018 Preliminary Economic Assessment is intended to be used by Deep-South, subject to the terms and conditions of its contract with the Report Contributors. Recognizing that Deep-South has legal and regulatory obligations, the Report Contributors have consented to the filing of the Haib Copper Project 2018 Preliminary Economic Assessment with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval ("SEDAR").

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1. SUMMARY

1.1 Property Description, Location, Ownership and Access

This independent Preliminary Economic Assessment ("PEA") Report has been prepared at the request of Deep-South Resources Inc (TSXV – DSM) ("DSM or The Issuer") which is listed on the TSX Venture Exchange (the "Exchange"). DSM has a 100% interest in Deep South Mining Company (Pty) Ltd. a Namibian subsidiary which has a 100% interest in Haib Minerals (Pty) Ltd ("HM") which in turn holds the exploration rights to the Haib Porphyry Copper property in the Karas Region, southern Namibia. Teck Resources Limited ("Teck") owns 35% of DSM. HM is the registered holder of Exclusive Prospecting Licence 3140 ("EPL") over the property.

The Haib deposit is a large palaeo-Proterozoic copper porphyry deposit, located in southern Namibia. Access to the Haib project camp-site is via a 10km graded gravel road from the main Cape Town – Windhoek north-south tar road and is accessible to conventional cars. Access from the camp-site to the main deposit area is along some 5km of tracks suitable for four-wheel drive vehicles. The site is very rugged and there is only limited access along numerous bulldozed tracks. The project area borders on both a summer and winter rainfall area, is very arid and in summer the temperature can go as high as the mid 40^0C, while in winter it can go as low as freezing point. Average annual rainfall is 25-50 mm.

The main Haib deposit straddles the Volstruis River, a tributary of the Haib River, which is an ephemeral tributary of the Orange River which lies some 10-15 km south of the deposit. The deposit has a distinct surface expression with abundant copper staining on fractures and joint planes particularly in and around the dry river bed of the Volstruis River. It was discovered in the late 1800's / early 1900's.

1.2 History and Source of Data

This report is based on a review of historical and currently available data concerning the Haib property obtained from HM and from Mr. V. Stuart-Williams, who is familiar with the project based on numerous visits and direct involvement as geological consultant to the Namibian Copper Joint Venture ("NCJV") in the period 1995 to 1999 and since 2004 as the current Technical Director of DSM. The sections and paragraphs which describe the estimation of Resources at the Haib are the work of Mr. D.S. Richards of Obsidian Consulting Services [(1)].

In 2008 DSM entered into a contract with Teck which offered Teck an option to obtain a major interest in the project in return for meeting certain exploration expenditure commitments. Teck fulfilled all of its commitments under the agreement and as a result 70% of the shares in HM were issued to Teck. In May 2017, Teck agreed to transfer ownership of their shares in HM to DSM in exchange for a 35% interest in DSM.

1.3 Exploration Rights

The exploration rights and obligations over the Haib property are held by HM under EPL 3140. This licence originally had an area of 74,563 ha and it incorporated all of the known mineralisation within the Haib deposit and a substantial area around the deposit. The EPL was renewed in April 2007, April 2009, April 2011, April 2013, April 2015 and April 2017. The current area, after reductions required in terms of the first renewal, is some 36,502 ha and the licence is valid until April 2019 (see Appendices 1, 2 & 3).

1.4 Geology and Mineralisation

The Haib deposit is located within part of the Richtersveld geological province [36]. The area lies within the Orange River Group volcanic suite of andesitic lavas, intercalated with acidic volcanics and tuffs, which were intruded by Vioolsdrif Intrusive Suite granites, granodiorites and adamellites dated around 1,880 Ma. The principal mineralised hosts at the Haib are a Quartz Feldspar Porphyry (QFP) and a Feldspar Porphyry (FP). The entire sequence has undergone low grade regional metamorphism to greenschist facies. At the Haib there is a further overprint exhibited by typical porphyry copper type alteration zones associated with this style of mineralisation.

The Haib copper deposit is an example of a Precambrian porphyry copper deposit. Porphyry copper deposits are a major world source of copper. It is in essence a very large volume of rock containing low-grade copper mineralisation.

1.5 Exploration, Drilling, Data Verification and Quality Assurance and Control

Five separate geophysical, geochemical and diamond core-drilling exploration programmes have been conducted at the Haib by Falconbridge (eleven drillholes); King Resources (twenty-one drillholes); Rio Tinto Zinc (“RTZ”) (one hundred and twenty drillholes); the NCJV / Great Fitzroy Mines (“GFM”) joint venture (12 drillholes); and Teck / HM (32 diamond drillholes).

The Haib Project drill hole database comprises data from a number of exploration phases completed over an extended period of time by multiple companies. Most of this work preceded the introduction of mineral resource reporting guidelines such as CIM, NI 43-101, JORC and others and as such are not supported by the requisite Quality Assurance and Quality Control programmes. While it is assumed this work was done to the international standards of the time, the work done by Teck from 2008 onwards utilised a full formal QA\QC programme and included the resampling of a significant portion (~8%) of the available RTZ core. Comparison of the Teck data to the historical data for the same sampled volume shows identical Cu grade distributions confirming, in the author’s opinion, that the historical data is of a suitable quality for use in mineral resource estimation and reporting along with the Teck data.

1.6 Mineral Resource Estimation

In July 2017, Obsidian Consulting Services was contracted by DSM to compile a resource estimate for Copper (“Cu”) and Molybdenum (“Mo”) from various previously identified and modelled domains for their Haib Copper Project [(1)]. The assignment included the compilation and validation of a drill hole database derived from historical core drillholes and more recent core drillholes and check assays of historical core samples completed over the past 9 years by Teck Namibia.

Estimation was done using Ordinary Kriging in a step-wise manner. A first kriging run was done using a search equal to the variogram ranges. A second pass was then done using a search double the variogram ranges. Almost all cells were estimated by the above 2 steps. Any remnants were either kriged by opening the search and increasing the minimum samples or assigned a background value such as the median.

A boundary analysis showed a mineralisation trend that correlates with the mineralisation model of the Haib deposit viz. Cu porphyry. The changes in grade are generally gradational and cross lithological contacts therefore the use of lithological boundaries was felt to be inappropriate while the use of a grade cut-off based domain was considered arbitrary. Therefore, no domaining was applied in the mineral resource compilation with the extents being controlled by data proximity.

Ten metre composites were calculated from the drill hole data which approximates a horizontal grid of approximately 150m x 150m. Analysis of the composites showed that compositing does not fundamentally change the Cu grade distributions. Top cut analysis of the Cu grades showed the raw data to be relatively insensitive to the use of grade caps and no capping was applied during the compositing. The same is not true for the Mo dataset which is substantially smaller than that of Cu and includes qualitative visual estimates alongside quantitative analytical results. Confidence in the Mo data is therefore very low compared to Cu.

Variography revealed the presence of well supported spherical variogram models with anisotropic elements. The Cu values show that the maximum anisotropy is oriented along a strike of 135º dipping to the southwest at just under 60º. The Mo values show similar levels of anisotropy with a principal continuity direction of some 207º. Estimates were conducted in a stepwise manner first at the modelled Ranges followed by a second estimation run at 2x the Ranges. In the case of Cu, 99% of the block model cells were estimated within the Ranges while for Mo this was some 85%. Anomalously high grades were used as is but had their volume of influence significantly reduced to avoid overestimation and the introduction of bias. During estimation, the quality of the estimates was tracked using kriging parameters such as kriging variance, slope of regression and kriging efficiency. Post-estimate validations were done use QQ Plots, Swath Plots and comparison with nearest neighbour and inverse distance estimates.

Density was not estimated but rather an average of 2.8T/m^3 taken from the mean of the 99 Specific Gravity determinations that had been done. These showed a normal distribution around a mean of 2.75 T/m^3 with a very low coefficient of variation (0.04) therefore the default density approach is considered appropriate.

The mineral resources for Haib have been classified according to the guidelines of the Canadian National Instrument 43-101 and is based primarily on proximity to data with the last line of samples forming the lateral and vertical extent of the resources. Secondary considerations included slope of regression and kriging efficiency. In our (Peter Walker and Dean Richards) opinion and considering the current price of copper we assume that a 0.25% Cu cut-off can be mined and processed economically and trust that the recently commissioned Preliminary Economic Appraisal Report will confirm this assumption. This cut-off was applied to the mineral resources presented in Table 1-1 below.

Table 1-1: Haib Resource Estimates

Resource Class	Volume (xMillion m$^{3)}$	Density	xMillion Tonnes	Cu(%)
Measured	-	-	-	-
Indicated	163.2	2.8	456.9	0.31
M+I	163.2	2.8	456.9	0.31
Inferred	122.3	2.8	342.4	0.29
Rounding has been applied as appropriate to reflect limits of precision and accuracy				

1.7 Mineral Reserve Estimate

There is no NI 43-101 compliant reserves estimation for the Haib mineralisation.

1.8 Mining Methods

Considering the Haib copper deposit characteristics, the suitable mine design is based on an open pit method. As the deposit is basically composed by hard rock material, the mining operations will involve drill and blast of all excavated material, which will be segregated by cut-off grade.

1.9 Recovery Methods

For the recovery of copper from the Haib deposit, heap leaching was considered for all options. The primary reasons for the selection of heap leaching is the low grade nature of the deposit and the vast scale of the orebody. Previous work conducted on the Haib project suggests that a conventional crush-grind-float and sale of copper concentrate is not economically feasible due to the low grade and hardness of the ore – requiring a significant amount of energy for grinding. The low costs associated with heap leaching compared to a whole ore flotation circuit is believed to improve the viability of the project. Heap leaching is traditionally performed on oxide material, although there has been increasing development in the application to acid insoluble sulphides. Previous sighter amenability testwork suggests the Haib material can extract high amounts of copper, up to 95.2% via a bacterial assisted leaching, although additional testwork is required to determine the optimal operating parameters. Given these results there is no reason to suggest the chalcopyrite in the Haib deposit will not be amenable to bacterial assisted heap leaching.

Four options were established for the purposes of the economic evaluation:

- Option 1: Ore sorting upgrading, dense media upgrading, flotation and heap leaching of the tails;
- Option 2: Two-stage dense media upgrading, flotation and heap leaching of the tails;
- Option 3: Ore sorting upgrading and heap leaching of the upgraded material; and
- Option 4: Whole ore heap leaching.

All options include molybdenum recovery, although the operating costs, capital costs and revenue have not been included, and will be considered if the molybdenum progresses into indicated resource.

1.10 Project Infrastructure

The following infrastructure considerations were made for the Preliminary Economic Assessment:

- Mine power;
- Mine buildings;
- Explosive storage;
- Waste dumps;
- Tailings storage;
- Power supply;
- Water supply;
- Water management;
- Telecommunications;
- Buildings;
- Roads;
- Air services;
- Railways; and
- Ports.

1.11 Market Studies and Contracts

The proposed recovery methods are expected to produce all or a combination of the following products:

- LME copper, as 99% cathode sheets;
- Copper sulphate, in the form of pentahydrate crystals, which is expected to generate a 25% copper premium;

- Copper concentrate, which is expected to be relatively high grade (33% copper) due to several upgrade stages, and therefore readily saleable; and
- Molybdenum trioxide, in powder form (only considered in process design, not the financial components).

1.12 Environmental Studies, Permitting, Social and Community Impact

A future environmental study will be required to assess:

- Baseline study;
- Environmental management plan;
- Project environmental assessment;
- Environmental issues (dust, noise etc.).

1.13 Capital and Operating Estimates

In summary, the capital and operating costs for the four options assessed are summarised in Table 1-2 and Table 1-3.

Table 1-2: Capital cost summary.

Cost (US$M)	Option 1	Option 2	Option 3	Option 4
Crushing & HPGR	54.3	50.0	56.2	53.3
DMS & Grinding	13.7	23.4	-	-
Flotation	3.0	3.3	-	-
Agglomeration & Heap Leaching	12.5	14.7	12.4	20.4
Copper Recovery	31.6	35.1	32.1	38.0
Iron Removal	1.6	1.9	1.8	2.7
Tailings	1.7	2.2	-	-
Water	2.9	3.7	2.8	2.8
Reagents	2.7	2.0	1.6	1.9
Services	2.0	2.0	2.0	2.0
Sulphuric Acid Production	21.4	27.1	22.0	27.3
Supporting Infrastructure	2.8	2.8	2.8	2.8
First Fill	10.8	13.0	6.0	8.3
Working Capital	16.1	18.1	14.0	16.0
Insurance	3.8	5.4	3.3	4.8
EPCM	16.1	18.1	14.0	16.0
Contingency	16.1	18.1	14.0	16.0
Commissioning	3.2	3.6	2.8	3.2
Accommodation & Temp Services	3.2	3.6	2.8	3.2
Spares & Tools	1.7	2.0	1.5	1.7
Total (US$M)	221.2	250.1	191.8	220.3

Table 1-3: Operating cost summary.

Area	Option 1	Option 2	Option 3	Option 4
Mining	0.40	0.37	0.41	0.37
Processing	0.88	0.92	0.82	0.83
Product Freight	0.05	0.05	0.04	0.04
Wharfage & Shiploading	0.01	0.01	0.00	0.00
Administration	0.04	0.03	0.04	0.03
Royalty	0.09	0.09	0.09	0.09
Total (USD/lb CuEq)	1.46	1.47	1.41	1.37

1.14 Economic Analysis

Based on the economic analysis of four options, Option 3 – an upgraded heap leach via ore sorting – was considered the best economically. The summary for the base case (US$3.00/lb copper) for each option can be seen in Table 1-4.

Table 1-4: Economic summary.

Financial Metric	Option 1	Option 2	Option 3	Option 4
Throughput (Mtpa)	8.5			
Copper Recovery (%)	77.1	82.1	73.2	80.0
CAPEX ($M)	US$221.2	US$250.1	US$191.8	US$220.3
Total Operating Expense ($/lb Cu.Eq.)	US$1.46	US$1.47	US$1.41	US$1.37
$NPV_{7.5\%,\ pre\text{-}tax}$ ($M)	US$645.1 (CA$817.6)	US$662.6 (CA$828.3)	US$716.2 (CA$895.3)	US$794.1 (CA$992.6)
$IRR_{pre\text{-}tax}$ (%)	25.9%	24.4%	30.4%	29.7%
Payback Period $_{pre\text{-}tax}$	5.0 years	5.3 years	4.2 years	4.3 years
$NPV_{7.5\%,\ post\text{-}tax}$ ($M)	US$421.0 (CA$526.3)	US$434.3 (CA$542.9)	US$463.1 (CA$578.9)	US$514.1 (CA$642.6)
$IRR_{post\text{-}tax}$ (%)	20.0%	19.0%	23.0%	22.6%
Payback Period $_{post\text{-}tax}$	6.7 years	7.1 years	5.7 years	5.8 years

There is an opportunity to increase the throughput of the production in order to improve the project economics. It is suggested to stagger the increased throughput over time to ensure there is no over capitalisation and target production figures can be met prior to expansions. A scenario at 20 Mtpa for

Option 3 was performed, with the key economic outcomes observed in Table 1-5.

Table 1-5: Option 3 at an increased 20 Mtpa throughput.

	8.5 Mtpa Scenario ($3.00/lb Cu)	20 Mtpa Scenario ($3.00/lb Cu)
CAPEX	US$191.8M	US$320.5M
Total Operating Expense[1]	US$1.41/lb CuEq	US$1.44/lb CuEq
$NPV_{7.5\%,\ pre\text{-}tax}$	US$716.2M (CA$895.3M)	US$1,366.8M (CA$1,671M)
IRR $_{pre\text{-}tax}$	30.4%	38.6%
Payback Period $_{pre\text{-}tax}$	4.2 years	3.3 years
$NPV_{7.5\%,post\text{-}tax}$	US$463.1M (CA$578.9M)	US$854.9M (CA$1,061.9M)
IRR $_{post\text{-}tax}$	23.0%	28.6%
Payback Period$_{post\text{-}tax}$	5.7 years	4.5 years
LOM	55 years	24 years

[1]Higher unit operating cost for 20 Mtpa due to capped copper sulphate production (the equivalent copper pounds are not directly scaled due to the lower portion of the premium product).

1.15 Interpretation and Conclusions

In our (Peter Walker and Dean Richards') opinion, the Issuer is exploring a large volume porphyry copper deposit situated in an ideal location adjacent to modern infrastructure which has the potential to become a large copper producer. There already exists a significant body of technical data concerning the Haib mineralisation and the period between resource estimation, pre-feasibility and definitive feasibility studies could be relatively short.

METS believes the low grade nature of the Haib porphyry copper deposit makes it an ideal candidate for heap leaching. Developments in heap leaching of refractory ores in conjunction with modern ore sorting technology have the ability to maximise the economic potential of the Haib project.

1.16 Recommendations

DSM has commissioned this PEA which uses the data from the recent resource estimate (1) and this PEA will guide future expenditure on more detailed assessment of the project.

HM, as the operator of the Haib project has proposed to the Ministry of Mines, as motivation for renewal of the EPL, a programme of exploration over the next 18 months involving the recently completed Resource Estimation study, a PEA, a 1: 10,000 scale geological mapping exercise of outlying areas of interest, further metallurgical tests, geotechnical drillholes to aid advanced mining studies including a planned open-cast mine pit, and more detailed resource estimates and economic studies. These programmes are estimated to cost US$ 625 000 (currently equivalent to some C$ 800 000 or N$8.0 million).

In our opinion this programme has real merit and it is recommended that HM proceed with the proposed programme.

In METS opinion, the results from the PEA have been promising, and going forward METS recommends Deep-South Resources conduct a Pre-Feasibility Study (PFS). To improve confidence in the PFS results a more detailed metallurgical testwork will be required to validate the process options outlined in this report.

2. INTRODUCTION

2.1 Scope of Work, Terms of Reference and Purpose of the Report

The mandate given to the authors is to provide the Board of Directors of DSM ("The Board") with an updated independent technical review of the Haib property which incorporates the Resource Estimate [(1)] and the PEA and to comment on the efficacy of the further exploration programme proposed by HM. The report is to be used by DSM in a capital raising exercise.

This independent PEA has been prepared at the request, on 4th September 2017, of Mr. P. Léveillé, President and CEO of DSM. The fee for the preparation of this Report is being paid by DSM and is not dependant on the outcome of any capital raising exercise.

2.2 Principal Sources of Information

The Report was completed by the Contributing Authors and relies extensively on information, materials, representations and exploration data provided by historical and more modern data records obtained from The Boards, Teck and Mr. V. Stuart-Williams, a Professional Geologist registered with the South African Council for Natural Scientific Professions, registration No.400266/87, who has been associated with the Haib Project in various technical and advisory roles for over 20 years and is currently the Technical Director of DSM. The principal author has incorporated sections from the report of P&E Walker Consultancy cc and Obsidian Consulting Services ("Obsidian") dated 14th January 2018 in writing this PEA report and is not responsible for the results, estimates or conclusions of these sections.

This Report has undergone extensive review by The Board, their advisors and by the contributing authors to ensure that the information and representations contained in the Report are current, accurate, correct and complete and that there are no material omissions of information that would affect the conclusions contained in the Report.

The PEA Report is to be read as a whole and sections or parts of it should not be read or relied upon out of context. This notice, which is an integral part of the Report, must accompany every copy of the Report.

This entire Report is subject to the scope of work conducted as well as the assumptions made and to all other sections of this Report.

The effective date of this report is 3rd March 2018. The Qualified Persons and authors of this Report and their business entities have no direct or indirect interest in the subject or any nearby mineral property and are entirely independent of DSM and its shareholders.

2.3 Site Inspections

Peter Walker visited the Haib Project site described in this report on various occasions between 1989 and 1995 and on the 24th January 2012 in the company of Mr. Nuri Ceyhan, exploration manager of Teck Namibia and with

Mr. Neil Grumbley, Teck's Haib Project manager and again on the 30th June 2015 with Mr. Neil Grumbley. Peter is assured by the HM management that as at the date of this report no further field work or material change has occurred at the Haib project site since my June 2015 visit and that only desk-top appraisal studies as outlined in this report have been concluded since that visit.

Damian Connelly visited the Haib site in 2006. The objective of the site visit was to assess the surrounding infrastructure, view drill core samples and obtain a general feel for the site. No site visit was undertaken for the PEA.

3. RELIANCE ON OTHER EXPERTS

3.1 Other Experts

DSM requested that the principal author review the Haib Porphyry Copper Project and prepare a PEA of the project which incorporates the findings of the recent Technical and Estimated Mineral Resources report.

This report has been prepared under the guidelines of National Instrument 43-101 and is to be submitted as a Preliminary Economic Assessment Report to the TSX Venture Exchange ("TSX") in support of DSM's proposed capital raising.

The principal author is reliant on the expertise of the contributing authors for the results, estimates and conclusions of the Technical and resource estimate report.

3.2 Mineral Tenure

Peter Walker has reviewed the records [42 & 50] of the Namibian Ministry of Mines and Energy last updated on 4th December 2017 and believes that the Exclusive Prospecting Licence is in good standing; furthermore, the Issuer has provided the author with a copy of the recently renewed EPL licence document dated April 2017 and we have also obtained a legal opinion of the validity of the licence which confirms this belief (see Appendices 1, 2 & 3).

The opinion of Peter Walker regarding the validity of HM's rights to EPL 3140 as presented in this report are wholly conditional upon the accuracy and completeness of the information supplied by those references named above. Peter Walker reserves the right, but will not be obliged, to revise this report if additional information becomes known to him subsequent to the effective date of this report.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 The Exploration Licence and HM's Rights and Obligations

In Namibia the Ministry of Mines and Energy grants an Exclusive Prospecting Licence ("EPL") in terms of section 48(4) of the Minerals (Prospecting and Mining) Act, No.33 of 1992 to an applicant under certain terms and conditions which form part of the licence documentation – see Appendices 1 & 2.

Deep South Mining (Pty) Ltd obtained EPL No. 3140 allowing for the exploration of Precious and Base Metals and Base and Rare Metal Groups of Minerals over an original area of 74 563 ha on 22nd April 2004; the area extended over the known mineralisation of the central Haib deposit and a substantial surrounding area. The EPL was renewed in 2007, 2009, 2011, 2013, 2015 and again in 2017 and is valid to 22nd April 2019. In April 2007, the extent of ground held was reduced in accordance with the renewal obligations to an area of 36 502.4ha. The Table below lists the corner co-ordinates of the reduced EPL: -

Table 4-1: List of corner co-ordinates in decimal degrees for EPL 3140.

Licence	Nr	Lat	Long
EPL-3140	1	-28.72530758	17.78740212
EPL-3140	2	-28.71183732	17.74992010
EPL-3140	3	-28.67100381	17.68364770
EPL-3140	4	-28.62124926	17.75614951
EPL-3140	5	-28.55535382	17.82202069
EPL-3140	6	-28.62443708	17.92656044
EPL-3140	7	-28.62670927	17.95497900
EPL-3140	8	-28.69475300	17.96558957
EPL-3140	9	-28.72089474	17.96970697
EPL-3140	10	-28.75324360	17.95375930
EPL-3140	11	-28.74086885	17.79335440

Details of the location are given in the Location Map, Figure 4-1, while Appendices 1 and 2 are copies of the documents granting and renewing the EPL as well as recording the transfer of rights to the EPL from Deep South Mining (Pty) Ltd to HM.

The surface rights of the property covering portions of the farms de Villierspunt 353, Tsams 360 and Withoek 387 are owned by the State. The

EPL boundaries have not been surveyed or physically beaconed but the current corner coordinates have been provided by the Namibian government in the EPL grant documents.

Peter Walker isnot aware of any environmental obligations or liabilities except those listed in Part 3 of the attached Appendix 1 which states: -

“8. The holder of the exclusive prospecting licence shall observe any requirements, limitations or prohibitions on his or her prospecting operations as may, in the interests of environmental protection be imposed by the Minister from time to time.

9. The holder of the exclusive prospecting licence shall enter into an Environmental Contract with the Ministry of Environment and Tourism and that of Mines and Energy within one (1) month of the date of issue of the licence.”

The Environmental Contract and clearance certificate has been concluded with the respective Ministries and Peter Walker has had sight of these documents, which are dated 15 August 2017 and are valid for 3 years. Peter Walker has also been provided with a copy of the Environmental Management Plan produced by SLR Consultants [(43)] which was submitted in support of HM’s application to the Ministries and which now forms part of the accepted commitment towards HM’s environmental obligations.

Peter Walker is not aware of any additional permits required in respect of exploration activities on the property apart from water abstraction permits that will need to be obtained from time to time from the Ministry of Water Affairs in order to pump water from the Orange River for drilling purposes. Peter Walker has been provided with a copy of the last water abstraction permit which was valid from 28th March 2014 to the 27th March 2017 and that the only significant conditions attaching to this permit are for the installation of an approved water meter, monthly readings of the meter and payment at a tariff rate of 1.5 Namibian cents per cubic metre of water consumed.

As the subject property is State land, no access permits or contracts are required in terms of the grant of the EPL (see Appendix 1).

In order to retain title to the EPL, HM have to spend at least 80% of the committed budget for the 2017 / 2018 work programme which has been agreed with the Namibian Ministry of Mines & Energy (see also the conditions of grant as specified in Appendix 1); As at 14th December 2017, Peter Walker was assured by a letter received from the Directors of HM that their current expenditure on the project will meet and possibly exceed this minimum expenditure commitment. This letter also states that all obligations and requirements in regard to compliance with licence conditions have been met and that the future obligations of HM to report progress and abide by the agreed work programme have also been noted and will be complied with within the time frames agreed with the Ministry.

4.2 Location

The Haib copper deposit is in the extreme south of Namibia close to the border with South Africa which is defined by the course of the Orange River (see Figure 4-1). The deposit lies some 12 – 15 kilometres east of the main tarred interstate highway connecting South Africa and Namibia and the nearest railway station is at Grunau, some 120km north on the main highway. This rail connection could provide access to either the port of Luderitz or to Walvis Bay via Windhoek or to South African ports or facilities via Upington.

4.3 Project Ownership

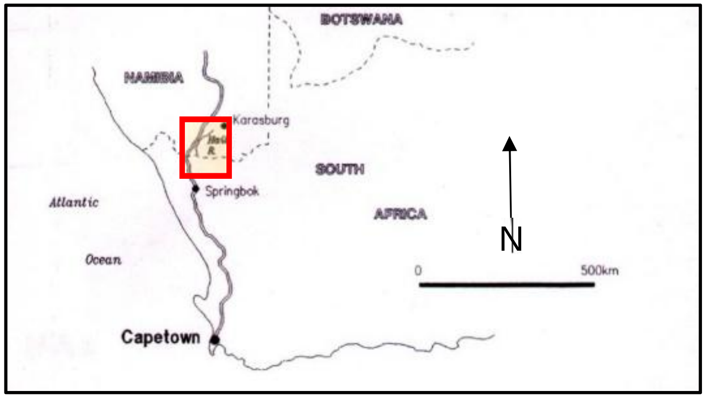
On June 20th 2008 DSM concluded a joint venture agreement with Teck, which was amended on March 9th 2009 (the "Agreement"). Teck then acted as the exploration operator and manager for HM.

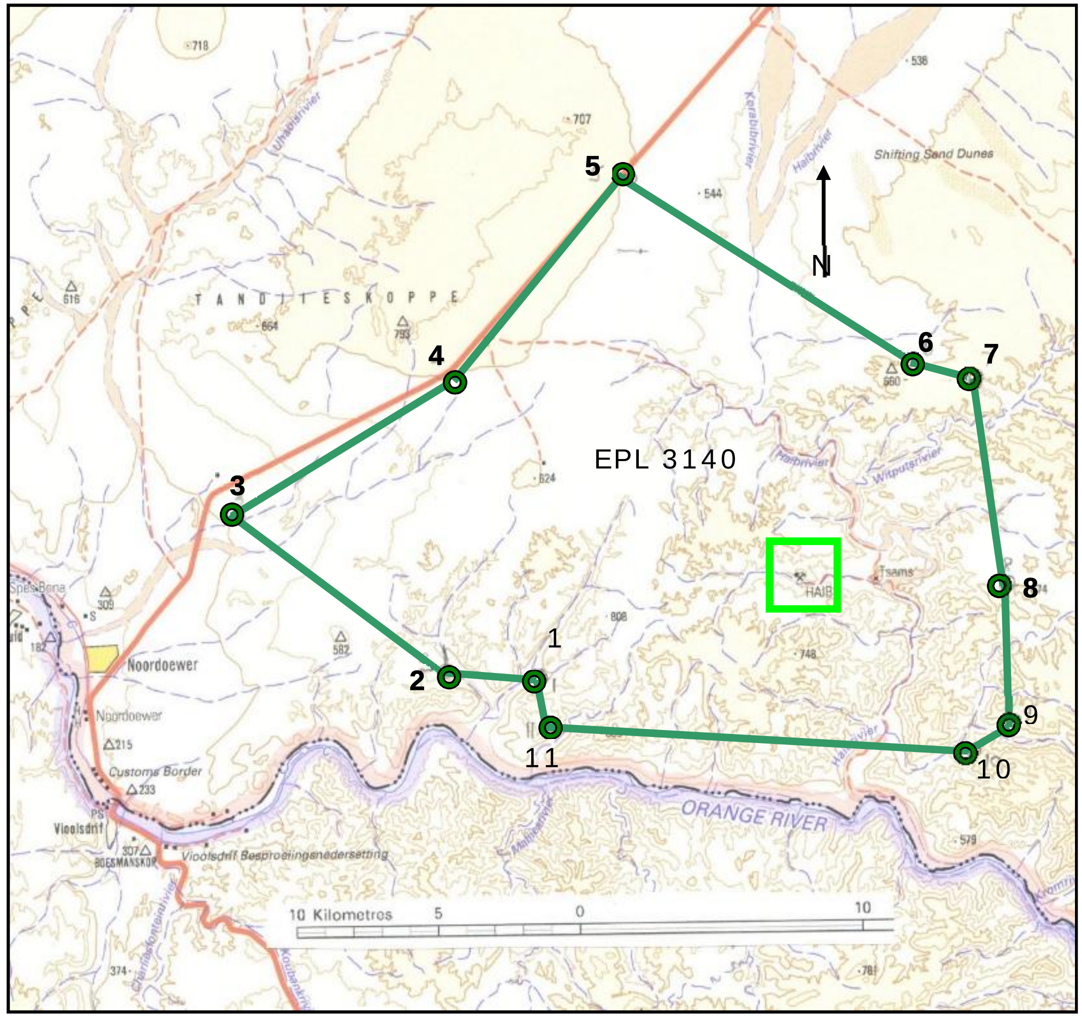
The Agreement with Teck provided that Teck had the right to earn a 70% undivided interest in the Haib copper project in Namibia if it completed an agreed programme of exploration which it duly complied with; Teck has now agreed to relinquish exploration management and its 70% interest in HM and has exchanged its rights and obligations in the project for a 35% shareholding in DSM.

The exploration approach taken by Teck was to prospect for adjacent, additional mineralisation by means of remote sensing, regional geophysical and geochemical stream and soil sampling programmes and / or to increase the tonnage and / or the grade by further core drilling to explore the already identified higher-grade portions of the mineralisation since these are poorly defined by the historical vertical drilling. Teck also completed an extensive programme of quality control and data checks by means of modern surveying of the historical drillhole collars as well as resampling and assays of a large number of the RTZ drill cores. The Teck exploration programme described in this report is the result of that exploration approach.

4.4 Contributing QP's Comment

I (Peter Walker) am not aware of any significant risk factors that may impede the progress of the exploration activities proposed for the property which may involve access, title or availability of contractors.





LEGEND

Location of the Main Haib mineralisation.

Figure 4-1: Location Map: The Haib deposit is situated some 12-15 kilometres east of the main tar road connecting South Africa and Namibia. Access from the tar road to site is via variable quality all-weather gravel road, the last section requiring all wheel drive. The current area of EPL 3140 is 36,502.4ha.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Physiography

The Haib deposit straddles the Volstruis River (meaning the Ostrich river in Afrikaans), which is a tributary of the Haib River. Both are ephemeral tributaries of the Orange River which lies south of Haib.

The Orange River is a deeply incised drainage with several nick-points. Haib lies below all of the main nick-points at a location where the Orange River elevation is approximately 200 metres above sea level. The Haib deposit lies at elevations from a floor elevation of just under 375 metres amsl to over 600 metres amsl. The surrounding area is up to about 650 metres amsl at the highest point. The area is rugged with steep sided valleys and rapid local relief.

The vegetation around the deposit is essentially xerophytic in nature with sparse semi-desert shrubs and grasses with some stunted trees (Adenolobus garipensis, Euclea pseudebenus or wild ebony[34] and others) along water courses.

5.2 Accessibility

Access to the Haib property is via a 10-km graded gravel road from the main interstate tarred highway to the camp site at the old Rio Tinto Zinc Corporation ("RTZ") exploration campsite. This road is accessible to conventional cars. From the campsite to the Haib copper deposit (another 5 km) is a four-wheel drive gravel track that is relatively slow but essentially all-weather. The site itself is very rugged and there is only limited access along the numerous bulldozed drill-site access roads. Access to other parts of the site is largely by foot. The topography of the site is illustrated in Photograph 1. There is an existing gravel airstrip, some 1,500m long on the property which is in unknown condition and would require inspection before light aircraft could safely make use of this facility.

Photograph 1: The photograph very clearly shows the rugged, barren nature of the area surrounding the Haib deposit. The view is looking northeast down the Volstruis River to the Haib River at the foot of the far hills. Almost all of the rocks visible in the foreground area of this photograph lie within the Main Haib deposit. The main access road can be seen running across the photograph from the photograph's bottom left hand corner. The white arrow indicates the location of the old workings in the Volstruis River and the nearby bulk sampling adit.

5.3 Infrastructure

The infrastructure in the area is good. The Haib deposit is relatively close to the main north-south tarred interstate highway between Cape Town and Windhoek so the only road construction required would be an upgrade to the existing ~12 km long access road to site. The nearest settlement is Noordoewer, some 12Km south of the Haib entrance gate, a village of some 5,000 people with only basic services and facilities.

The main north-south national power grid lines are some 85km to the east of the Haib; an 85-km link and upgrade of the line capacity would likely be required should the project be developed.

Water is currently available in large amounts from the Orange River which is about 15 kilometres by pipeline south of the main Haib deposit, however, future demand upstream may lessen the available water supply.

The nearest rail link is at Grunau station, some 120 kilometres north of the Haib. The area between the Haib and Grunau is almost completely flat and the local rail authority has confirmed that a link could be laid relatively easily; this would provide access to either the port of Luderitz or the port of Walvis Bay via Windhoek or to South Africa via Upington.

5.4 Climate

The Haib copper deposit is in the extreme south of Namibia and is unusual in that it is located on the boundary between the summer and winter rainfall areas. In summer the temperature can go as high as the mid 40^{0}C, while in winter it can go as low as freezing point. Rainfall in winter is generally light drizzle with occasional harder falls. In summer the rainfall is associated with occasional thunder storms and is of short duration, but can be of very high intensity. All of the streams within the area are ephemeral but can flow very strongly after summer storm rainfall. Average annual rainfall is 25-50 mm. Access to the site is possible throughout the year and there should be no interruptions to mining because of inclement weather [6, 30].

5.5 Sufficiency of Surface Rights

Suitable and sufficient areas for tailings dams, recovery plant, waste dumps and heap leach pads are available within the EPL area but the chosen sites will be dependent on the eventual mine and plant design. The project covers portions of the farms de Villierspunt 353, Tsams 360 and Withoek 387 which are State land and currently only used for emergency stock grazing purposes under lease from the State, so mining should not conflict with any formal farming activities.

6. HISTORY

6.1 Sources of Historical Exploration Data

The author (Peter Walker) draws his knowledge for this section from the Behre Dolbear [(2)] report; from the Namibian Copper Mines report [(3)], from the South African Committee for Stratigraphy (SACS) [(4)] and from the Gordon / McIlwraith report [(5)], and from personal knowledge. The author has only seen extracts of reports and third-party reports on the early mining at Haib and information referred to is gained from these reports and discussions with the late Mr. George Swanson (see below).

6.2 Early Mining

The Haib deposit has a distinct surface expression with abundant copper staining on fractures and joint planes, particularly in and around the dry river bed of the Volstruis River (see Photographs 1 & 2). This led to German prospectors identifying the deposit around the late 1800s or early 1900s. Small tonnages of high grade copper carbonate ore were mined at this time. Incidentally, the word Haib is probably derived from the local Nama language although the Haib Pforte (fort) is shown on the original German military maps of German South West Africa, dating from about 1907. The fort appears to have been a place rather than a structure and the location on the ground is unknown.

After World War II, the prospect was pegged as claims by prospector Mr. George Swanson who carried out small scale mining and tank leaching operations. Over 6,000 tonnes of hand sorted high-grade copper ore were sold to the O'okiep Copper Mines, across the border at Nababeep in South Africa, reportedly at grades of up to 18% Copper. Lower grade copper carbonate ore was leached with acid. The acid was then run over iron scrap and the copper precipitated as "copper cement". This copper cement was sold to the smelter at the O'okiep Copper Company for further refining. Swanson only worked these claims when the copper price was high enough to justify the process (personal communication).

6.3 Post-1960 Exploration [(2, 5 ,7)]

In 1963 and 1964 Falconbridge of Africa (Pty) Ltd ("Falconbridge") completed an exploration programme focused on the higher-grade zones within the Haib deposit. They drilled some eleven diamond drillholes totalling 1,012 metres of drilling. The average grade of the drillhole intersections was given as 0.33% Cu. In 1964 Falconbridge allowed their rights to lapse; very little of their data remains on record.

During 1968 and 1969 King Resources of South Africa Pty Ltd ("KRC") conducted a further diamond drilling programme of 21 holes totalling 3,485 metres. They examined both lower and higher-grade sulphide zones, as well as the higher-grade oxide shear zones. Some leach test work was carried out. The area was abandoned in 1969. Again, very little useful data survives from this programme.

During 1972 to 1975 Rio Tinto Zinc ("RTZ") conducted the first extensive and systematic investigation of the Haib deposit. Geochemical and chip sampling surveys were conducted along with IP and Resistivity surveys. They drilled one hundred and twenty diamond drillholes (120) totalled 45,903 metres, one section was partially drilled at 25 metres spacing to provide detailed information on close spaced variability (see Figure 6-1 below); the core from this programme is still intact and stored in a core shed on site (see photograph 3 & 4 below), although much of the mineralised sections are now reduced to quarter core. RTZ sampled by compositing half cores over 2 metre intervals and submitted these for determination of total copper and where appropriate, oxide copper (acid soluble copper). Composite samples from each drillhole were also tested metallurgically to determine recoverable copper and were assayed for molybdenum, silver and gold indicating average contents of 25 g/t Mo, 0.01 g/t Au, and 0.9 g/t Ag. Tonnage and grade estimates at various cut-offs were made and a conceptual pit design was proposed.

In 1991 and 1992 Revere Resources SA Ltd, produced a technical brochure and promoted the Haib as a "potential world class copper producer for the 1990s". It would appear that the intent was to list the company, possibly on the Johannesburg Stock Exchange, using the Haib as a property of merit. For reasons unknown to me (Peter Walker) this listing never materialised. No exploration work was done.

In November 1993 Rand Merchant Bank Ltd (of South Africa) ("RMB") acquired an option over the Haib property. Venmyn Rand Pty Ltd., mining management consultants to RMB then undertook a study of the project including compilation of all the available drillhole and assay records from previous investigations and set up a computerised drillhole database. It was concluded that the increase in the copper price since the 1970's, development of low cost / high tonnage mining systems and new and refined technologies such as bacterial leaching, solvent extraction and electro-winning combined to create a situation where development of the Haib deposit could represent an economic project; however, no further exploration work was done and work terminated in 1995.

In March 1995, Great Fitzroy Mines NL ("GFM") and RMB executed an Agreement in association with claim owner Mr. George Swanson to acquire 100% of the Haib project. GFM agreed terms with RMB whereby GFM could earn 90% of the project. Subsequently GFM agreed to transfer a 70% interest in the deposit to Namibian Copper Mines Inc. ("NCM") in exchange for NCM reimbursing past expenditure and providing GFM with a free 20% carried interest. NCM then purchased the remaining RMB interest leaving GFM with a 20% free carried interest and the management and NCM holding 80%(2). The operating company was called the Namibian Copper Joint Venture ("NCJV"). From 1995-99 the NCJV prospected the Haib managed by GFM. The names NCJV and GFM can be read as being synonymous.

Apart from the central mineralised core of the deposit which was covered by the Swanson claims and held under option by NCJV, the mineral rights over the greater Haib area were held by Copper Mines of Southern Africa (Pty) Ltd ("CMSA") as EPL 2152 and worked by the NCJV.

From 1995 to 1999 the NCJV drilled a further 12 infill holes, drilled 5 geotechnical investigation holes, completed 126 metres of excavation in an adit and two crosscuts for bulk sampling and metallurgical testing and carried out various test works including mining cost audits, bio-leaching studies, and milling and grinding studies. In February 1997 a Feasibility Study – Phase 2 Report was produced by their mining consultants, the Minproc – Davy Joint Venture [(17)]. The NCJV ran into financial difficulties and work was stopped at the Haib deposit in late 1998 to early 1999.

Rusina Mining Ltd of Perth, Australia acquired the concession from GFM / NCJV during 1999-2000 and they took over ownership of the Haib data. The transfer of the mineral rights to Rusina was apparently not ratified by the Namibian Government and no work was done by this company.

In 2003 (date uncertain) in response to the Namibian government enforcing the new Namibian Minerals Act, claim owner Mr. George Swanson, who throughout much of the Haib dealings had held some 69 x 18 hectare claims over the core area of the Haib deposit, was forced to finally relinquish his Haib claims as he had not prospected or mined the claims for some years. This meant that the property was free and 100% of the mineral rights were vested in the Namibian Government.

This allowed DSM to consolidate a single mineral rights entity over the entire Haib deposit. An initial Exclusive Prospecting licence 3140 was granted for 3 years from 22 April 2004 to 21 April 2007 over an area of 74,563.0 ha covering the deposit and a very large surrounding area. This was subsequently renewed in April 2007, 2009, 2011, 2013, 2015 and 2017 with the area reduced to 36,502.4ha after the 2007 renewal. The current EPL and details of its location are shown in Appendices 1 & 2.

Photograph 2:

This photograph is taken in the bed of the Volstruis River looking approximately south-west at the portal of the bulk sampling adit.

Photograph 3: View of the RTZ core shed in the background and the Teck drill core stacked in the foreground in metal boxes.

Photograph 4:
View of the RTZ core – split for assay and well preserved in wood & metal core trays.

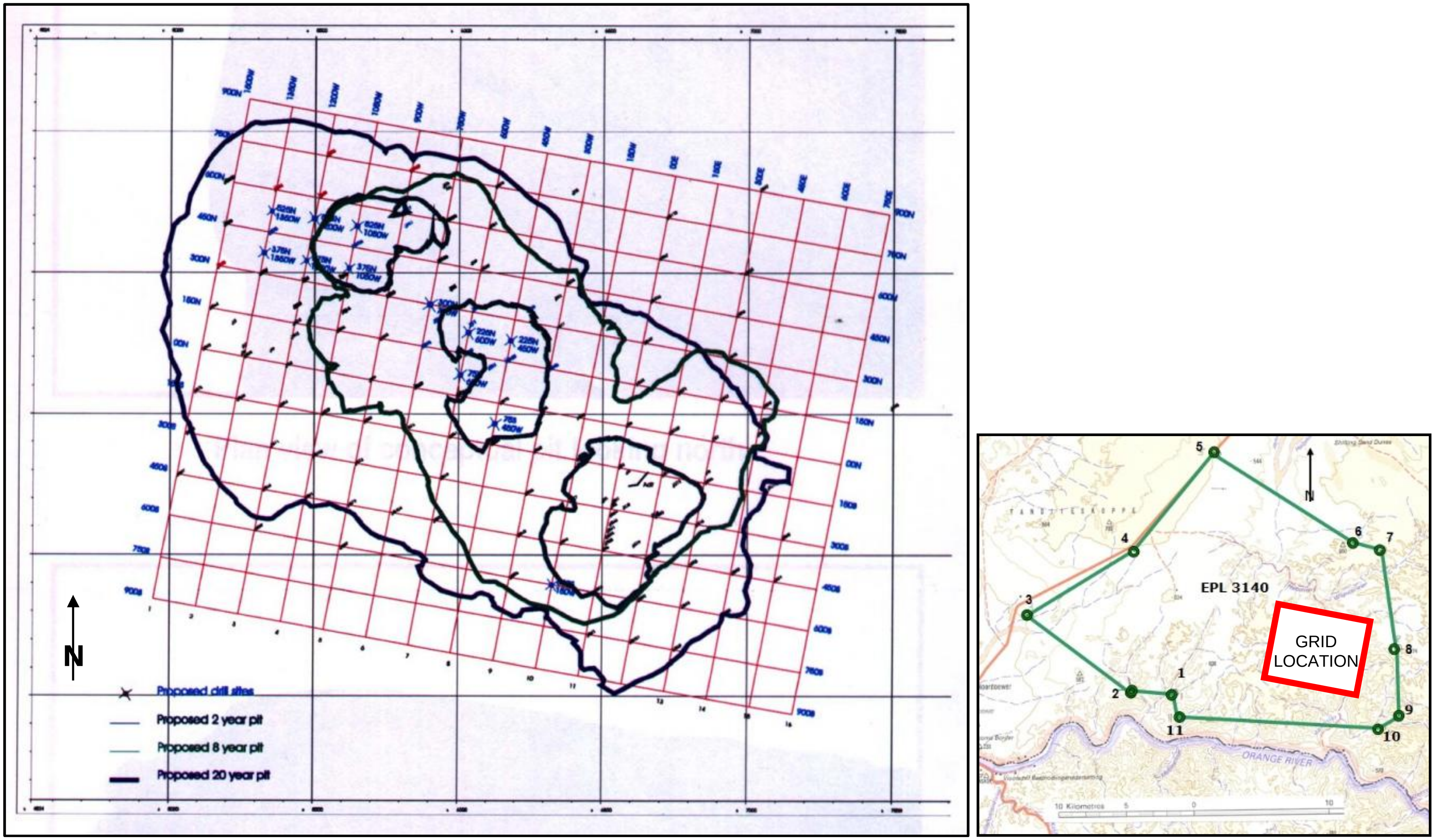


Figure 6-1: Proposed Mining Pit Layout: This is a reduction of a GFM map dated about 1996. The map shows the proposed 2-year, 8-year and 22- year pits generated from their geological model. The small black numbers indicate drillholes from the Rio Tinto and earlier drilling programmes. The larger blue numbers indicate drillholes that the NCJV / GFM proposed for drilling (June 1996). These drillholes were only drilled after the BD resource estimates were completed.

From 2008, Teck under the option Agreement with DSM had completed a comprehensive exploration programme at the Haib and immediate surroundings and it is the results of this programme that have justified the further studies incorporated in this report. DSM, through their wholly owned subsidiary, HM intend to continue to invest in an on-going exploration and development programme.

6.4 Historical Estimates

The tonnage and grade estimates quoted below are historic mineral estimates, that is to say they were prepared prior to DSM and subsequently, HM acquiring their interest in the Haib property. The estimates quoted here are therefore Historical Estimates as per the NI 43-101 Rules and Policies Part 1 definition of Historical Exploration Information.

I have not done sufficient work on the drill assay database nor have I (Peter Walker) had access to RTZ assay certificates or QA / QC data to classify the historical estimates as current mineral resources and neither Teck, DSM, HM or the Issuer are treating the historical estimates as current mineral resources or mineral reserves.

Four sets of historical estimates were prepared in the past by different authors. These will each be examined in turn. They are relevant in that they show the thinking of the investigators at that time and also provide insight into the areal extent and expected tenor of mineralisation.

6.4.1 RTZ Historical Estimate

Somewhere around 1975, RTZ, using the sample results from the 120 drillholes drilled by them (and perhaps the earlier drilling as well?), calculated an estimate of tonnage and grade for the Haib deposit. The figures reported suggest a very large volume of contained copper amounting to over 2 million tonnes of metal at a fairly low average grade of 0.27% Cu. RTZ used various cut-offs, but it is not reported what method of determination they used. The figures were considered by RTZ to be an Indicated Resource; however, they should be viewed as an Historical Estimate only (see Table 6-1)

Table 6-1: RTZ - Haib Historical Estimate

Cut-Off (% Cu)	Tonnage (Mt)	Grade (% Cu)	Contained Cu (t)
0.15	831	0.27	2,244,000
0.20	563	0.32	1,802,000
0.25	374	0.37	1,384,000

(Note: This is a Historical Estimate; a qualified person has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and the Issuer is not treating them as current mineral resources or mineral reserves)

Interestingly, RTZ seems to have concentrated on higher tonnages and not on the higher-grade zones. There is no evidence that they attempted estimates at any higher-grade cut-offs (such as 0.3% Cu). Clearly RTZ was interested in developing large volume mining resources.

6.4.2 Venmyn Rand Historical Estimate

In 1993 Venmyn Rand Pty Ltd prepared an information memorandum on the Haib deposit and estimated an in-pit "reserve" using a computer model, although the exact methodology is unknown. They generated the historical estimate presented in Table 6-2 below.

Table 6-2: Venmyn Rand – Haib Historical Estimate			
Cut-Off (% Cu)	Tonnage (Mt)	Grade (% Cu)	Contained Cu (t)
0.3	400	0.4	1,600,000

(Note: This is a Historical Estimate; a qualified person has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and the Issuer is not treating them as current mineral resources or mineral reserves).

The pit design used by Venmyn Rand was essentially conceptual and very large, being well beyond subsequent pit boundary designs. The Venmyn Rand estimate is thus considered to be effectively a global Historical Estimate above the 0.3% Cu cut-off.

6.4.3 NCJV / GFM Historical Estimate [3] (1996)

The NCJV used the Venmyn Rand computer database and recalculated their estimate around a more realistic geological and pit model. The pit model was designed to provide some 22 years of mineable material within a 2-year and 8-year mining pit plan. Geostatistical block modelling was carried out and tonnage and grades reported at a range of cut-offs within the various pit outlines. All drillhole assay results were composited over 7.5 metre down-hole intervals prior to variography and block kriging; the pit outlines were used to constrain the reporting of the block tonnes and grade which were thus reported as resource tonnages within a specified pit. The estimates were made in August 1996 and considered by GFM to be Indicated Resources although this category was chosen "...in accordance with accepted mineral industry practices" at that time.

The in-pit Historical Estimates as determined by GFM in 1996 (and approved by BD) based on the drilling to the end of 1975 are tabulated below in Table 6-3. Figure 6-1 above shows the proposed GFM two-year, eight-year and twenty-year pit outlines.

Table 6-3: GFM - Haib In-Pit Historical Estimate – June 1996							
Pit	Cut-Off 0.3% Cu		0.1%-0.3% Cu		Cut-Off 0.1% Cu		Waste
	Mt	% Cu	Mt	% Cu	Mt	% Cu	Mt
Year 2	21.4	0.39	27.9	0.20	49.1	0.28	2.1
Year 8	73.4	0.36	289.2	0.20	362.4	0.23	21.8
Year 22	135.5	0.38	803.4	0.19	939.1	0.22	95.7
Total	230.2	0.37	1120.5	0.19	1350.7	0.22	119.5

(Note: This is a Historical Estimate; a qualified person has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and the Issuer is not treating them as current mineral resources or mineral reserves)

6.4.4 Behre Dolbear Historical Estimate (2)

Behre Dolbear ("BD") viewed the Haib deposits as resources not reserves, because at the time of assessment they could not be demonstrated to be economic since no feasibility study had been completed. Therefore, BD undertook, after discussion with GFM, to review potentially mineable resources after the additional work had been completed, all or part of which could then be upgraded to a reserve status. This work was never completed. BD did not independently check the accuracy of the data provided by GFM but accepted the data as supplied for this work.

The drillhole data set provided to BD consisted of assay and survey data from 152 drillholes. The location of the drillholes was based on a local coordinate system. Included in the assay database were primarily the copper assays.

The historical mineral models generated by BD were estimated by generating three separate three–dimensional block models using nearest neighbour, inverse distance squared and kriging estimation techniques. Their results are compared with the GFM estimate in Table 6-4 below: -

Table 6-4: Haib Historical Estimate - Behre Dolbear / GSM								
Minimum Block Grade	GFM Model		Behre Dolbear's Model					
			Kriging		Inverse Distance Squared		Nearest Neighbour	
	M Tonnes	Grade % Cu	M Tonnes	Grade % Cu	M Tonnes	Grade % Cu	M Tonnes	Grade % Cu
0.1	1350	0.23	1353	0.23	1331	0.23	1184	0.25
0.2	730	0.28	739	0.29	726	0.29	630	0.34
0.3	230	0.37	244	0.37	262	0.38	292	0.46

(Note: This is a Historical Estimate; a qualified person has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and the Issuer is not treating them as current mineral resources or mineral reserves)

6.5 Contributing QP's Comments on the Various Models and Estimates

Both the GFM model and one of the BD models used Kriging methods as the basis for their estimate calculations. The figures generated are very comparable, the BD numbers being very slightly more generous in both grade and tonnes. Kriging as a statistical estimation technique is widely used in porphyry deposits.

When calculating block values with the inverse distance model, the arithmetic process decreases grade on an inverse basis away from the point of measurement. In simple terms this means that the grade initially decreases rapidly away from the data source then flattens off with distance. Parameters for the X, Y and Z axes are operator chosen and can be varied in different directions in the event of mineralisation being obviously controlled by any geological factor, such as faulting, unconformity contact or bedding.

The Haib deposit is not bedded, although some structural control over higher grade mineralisation is apparent, and has a fairly uniform grade distribution.

The nearest neighbour technique assigns the grade of the sample nearest the centre of the block to the block and provides a global check on the estimates.

It should also be noted from comments made elsewhere in this report during discussions of the recent Teck drilling that the historical, vertical drilling used for all of the above historical estimates may have incorrectly estimated both the extent and the grade of the high-grade zones because the high-grade zones lie within a dipping set of fractures and require inclined drillholes to obtain a true thickness estimate.

The most significant and well documented historical mineral estimate derives from the report by Behre-Dolbear [(2)] (1996) that was commissioned near the end of the NCJV tenure at the Haib and is summarized in Table 6-4 above. Please note: -

(1) That this estimate was prepared prior to publication of the National Instrument 43-101 guidelines and the CIM definitions and Standards for reporting of mineral reserves and resources in 2000 and their subsequent amendments in 2005, 2010 & 2014, and perhaps more importantly,

(2) The Historical Estimates developed by Behre Dolbear (1996) for the Haib deposit have been reviewed here by the author; however, the underlying data and evidence, particularly assay certificates, required for the author to validate and classify these Historical Estimates as current mineral resources are not available. Therefore, the historical grades and resources terminology from the historical original reports are to be used only as a reference and are to be considered as Historical Estimates as per the NI 43-101 Rules and Policies Part 1 definition of Historical Exploration Information. Neither DSM nor HM are treating the historical estimate as a current mineral resource or mineral reserve and do not rely on this estimate in any financial studies.

7. GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology [37]

The Haib deposit is located within part of the Namaqua-Natal Province called the Richtersveld geological sub-province which is further subdivided into a volcano-sedimentary sequence (locally, the Haib Subgroup), the Orange River Group and the intrusive Vioolsdrift suite which are closely related in space and time (Figures 7-1, 7-2 and 7-3 below). The Orange River Group is composed of sub-aerial volcanic rocks and reworked volcaniclastic sediments; deformation caused displacements along stratigraphic contacts before intrusion of the Vioolsdrift suite. The predominance of andesitic and calc-alkaline magmatic rocks with tectonic compression prevailing throughout the magmatic episode has led to an interpretation of an island-arc model for the region. Recent age dating of Haib rocks by separation of zircon and apatite on which laser ablation and inductively coupled plasma mass spectrography was used to derive the U/Pb ratios was performed at Trinity College, Dublin by Neil Grumbley and indicated an age of 1,880 Ma for the volcanics. [47]

The principal mineralised hosts at the Haib are a Quartz Feldspar Porphyry (QFP) and a Feldspar Porphyry (FP) – see Figures 7-2 and 7-3 below. The QFP is interpreted as a quartz diorite body which intruded the feldspar porphyry some 1,868 ± 7Ma [47]. The FP is generally interpreted as being part of the suite of andesitic rocks although some workers have suggested that it too, may be partially of intrusive origin. The QFP is elongated along the orientation of the Volstruis Valley, largely coincident with the location and orientation of many of the higher-grade intersections within the deposit.

The sequence has undergone low grade regional metamorphism to greenschist facies which event has been dated at 1,100Ma [47]. Most of the rock exhibits typical porphyry copper type alteration zones associated with mineralisation. A potassic hydrothermal alteration zone coincides with the main mineralised area surrounded by phyllic and propylitic alteration haloes. Propylitic and sericitic alteration appears to overprint the earlier potassic zones. Silicification, sericitisation, chloritisation and epidotisation are widespread.

Although not present in the immediate area of the Haib deposit, some kilometres to the west of the area are outcrops of Karoo age [4] (early Permian) mudstones, siltstones and sandstones of the Prince Albert Formation. These create very flat topography.

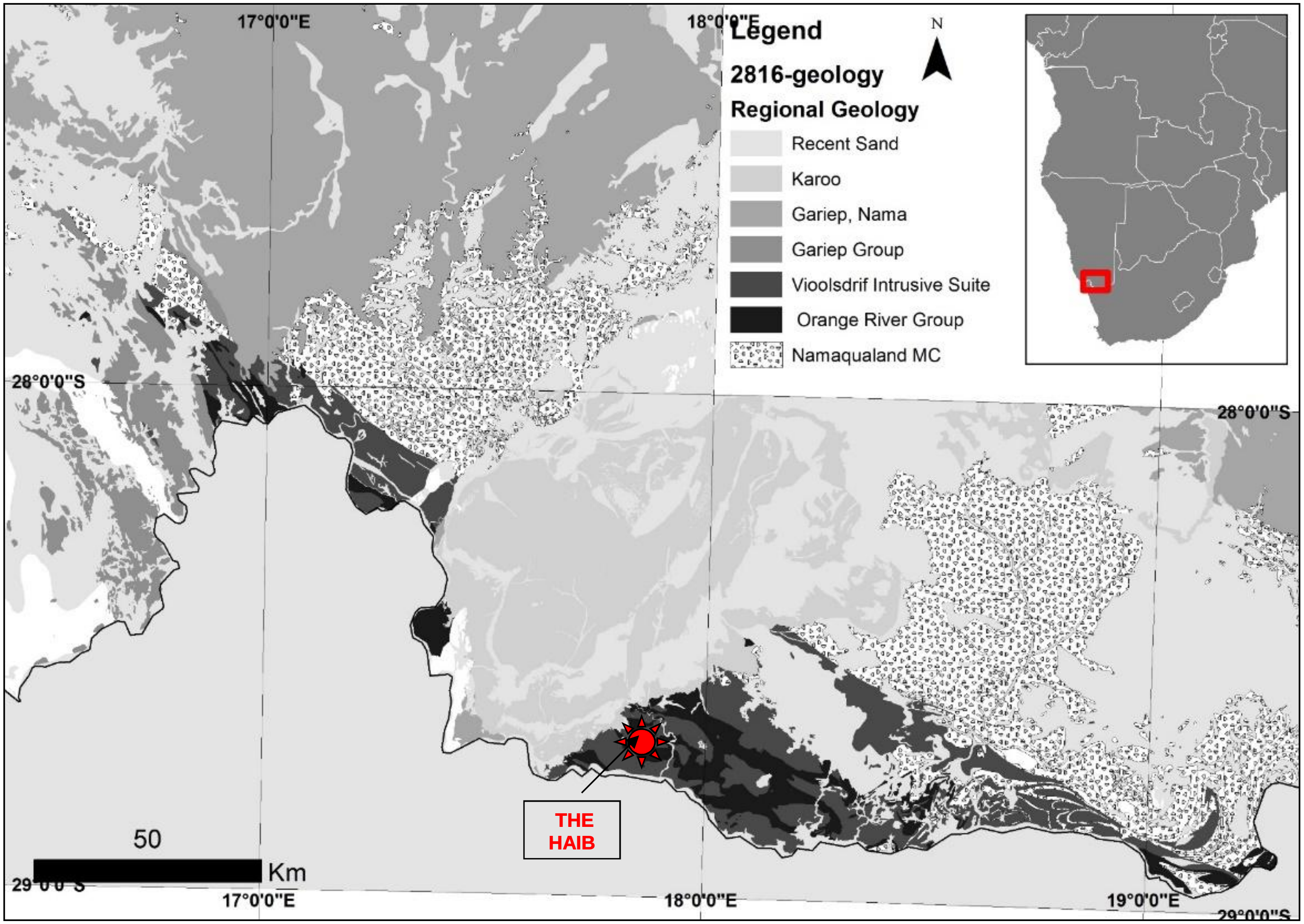


Figure 7-1: Regional Geology: Map showing the general distribution of the Vioolsdrift and Orange River rocks in relation to the Haib deposit. (Source: Teck Namibia [45], 2015).

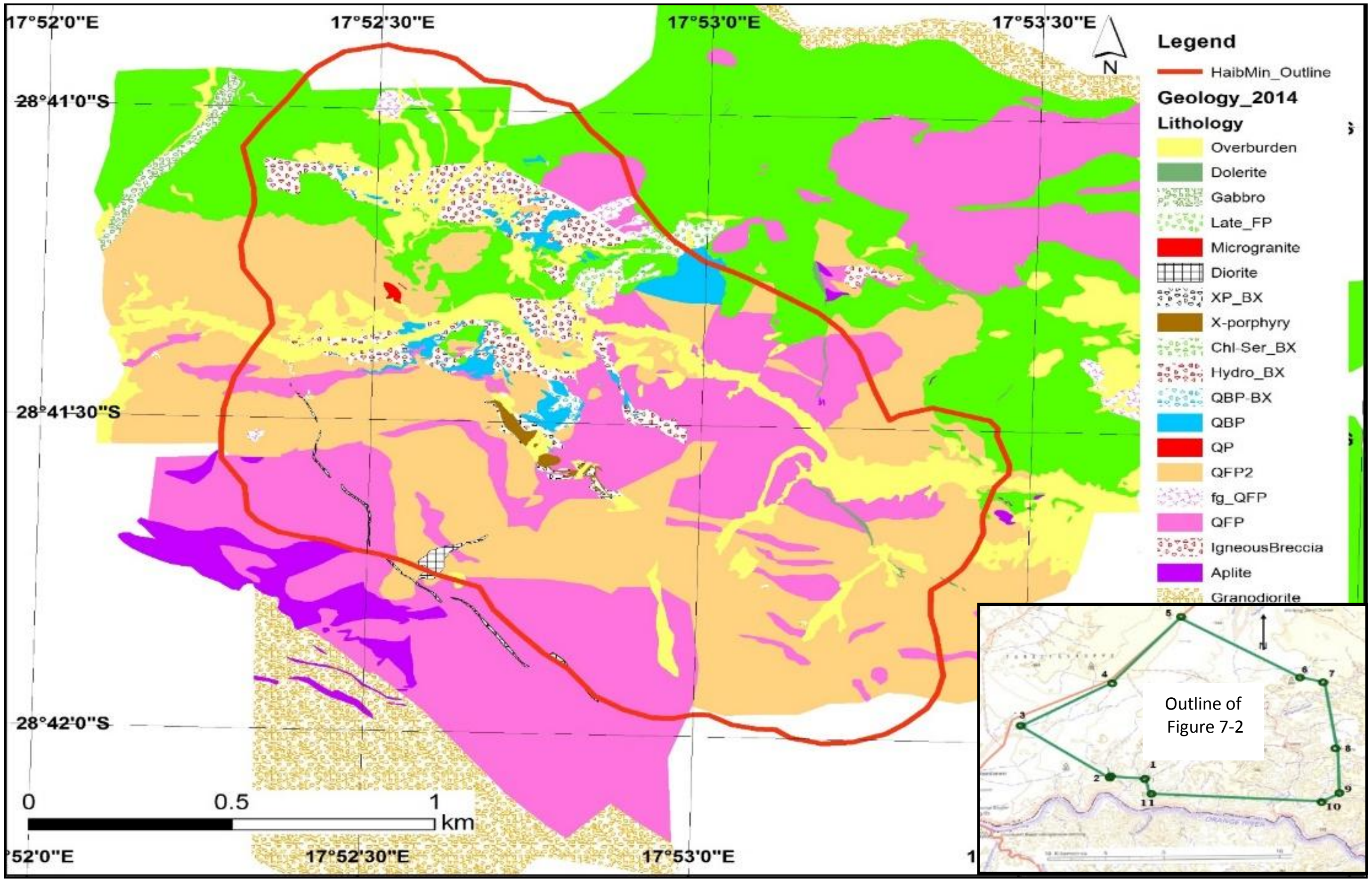


Figure 7-2: Local Geology: Geology of Haib (from Teck 2015 [45])

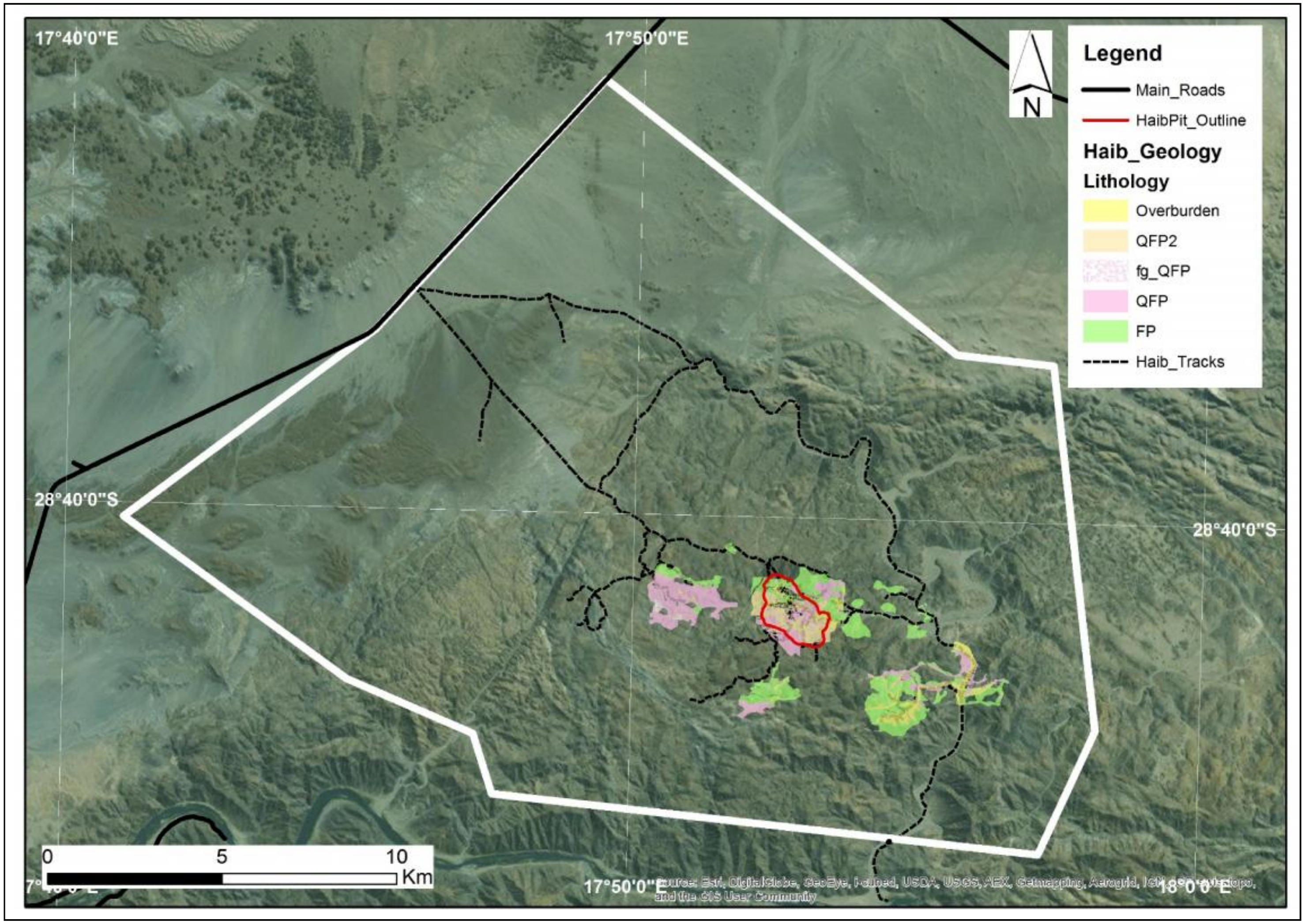


Figure 7-3: Intrusive Rocks: This map shows the detailed intrusive rock units of the Haib deposit. (Source: Teck 2015 [45])

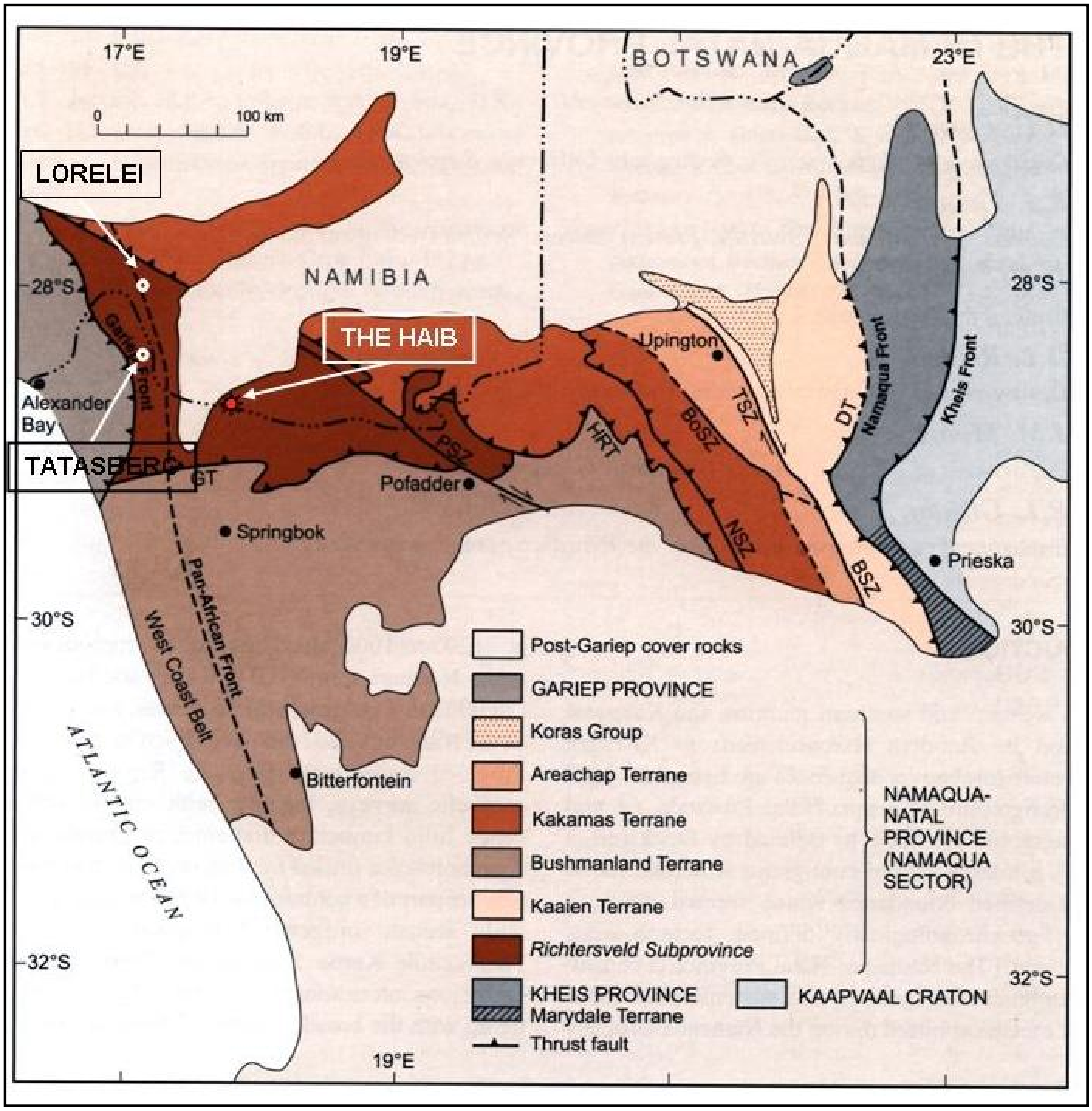


Figure 7-4: Regional Tectonic Map: Tectonic sub-division of the Namaqua sector of the Namaqua-Natal Province, as sourced from the Geology of South Africa, pg.326[37].

BoSZ: Boven Rugzeer Shear Zone, BSZ: Brakbosch Shear Zone, DT: Dabep Thrust, GT: Groothoek Thrust, HRT: Hartebees River Thrust, NSZ: Neusberg Shear Zone, PSZ: Pofadder Shear Zone.

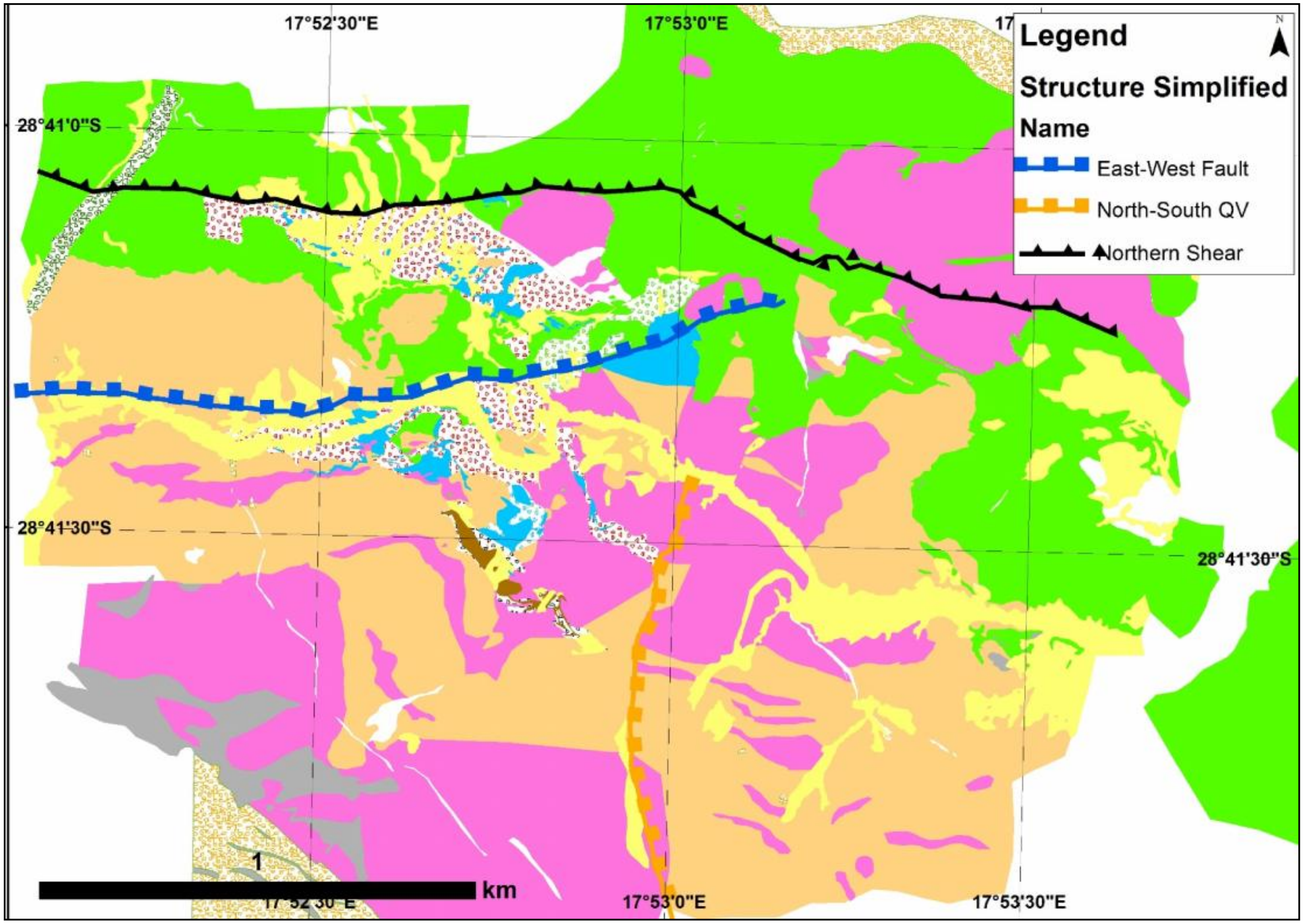


Figure 7-5: Local Tectonic Map: The three main structures recognised at Haib: 1) The Northern Shear Zone truncates porphyry mineralisation to the north, with reverse of movement; 2) the East-West Fault has normal movement down dropping mineralisation by >100m to the north; 3) the North-South Quartz Vein has normal movement, down dropping mineralisation by >100m to the SE. There are also numerous other smaller faults and shears with 1-5m displacement (Teck 2015 [46]).

7.2 Local Geology of the Haib Deposit [19, 20, 23, 45, 46]

The QFP comprise typically blue quartz and feldspar phenocrysts within a medium grained rock mass of quartz, feldspar, sericite, biotite, chlorite, epidote and calcite. The FP is generally a medium to fine grained rock of similar composition but without the quartz phenocrysts and with a higher proportion of chlorite and epidote; please note that the sericite, epidote, chlorite and calcite are alteration products and not the original igneous composition of the QFP or the FP (Figures 7-2 and 7-3 above). Minor basic dykes and quartz veins traverse the area.

Rocks within the Haib area are hard and competent but generally well jointed with both flat and steeply dipping joint sets being well developed. Striking east-west along the Volstruis River is a well-developed zone of steeply dipping shears. The orientation and location of the main mineralisation coincides with the fracture zone which is interpreted as fractures providing a focus of the intrusion and then channel-ways for late-stage mineralising fluids. The fracture zones likely represent the local stress regime at the time of porphyry formation and control the orientation of high grade zones, and were later re-activated by the Namaqua deformation event circa 1,100 Ma ago (Figure 7-5 above).

7.2.1 Structural Controls on Copper Mineralisation [45]

Mineralisation at Haib is typical of a porphyry copper deposit and despite the age of the deposit, and the fact that the mineralisation has been subjected to local post-mineral deformation, the deposit remains relatively intact. Detailed mapping by Teck geologists within the main deposit area has shown that high-grade copper mineralisation is controlled by a fracture/vein set that parallels a regional structural trend and strikes N60^{0}W and dips steeply (-70^0) to the southwest. This high-grade zone also appears to plunge at 30^0 to 40^0 towards the south-east (see Figures 7-5 above and 7-6 below). This model has significant economic implications as it suggests that the higher-grade zone of copper mineralisation has not been adequately tested by the historical vertical drillholes and that inclined drillholes will better define the extent and tenor of this mineralised zone. If this model is correct then systematic inclined drilling could better define the high-grade sections leading to better pit design to exploit near-surface high-grade mineralisation at the start of mining operations.

Teck has also defined four new target areas near to the main deposit and three other target areas on the property, namely the SW alteration feature, the NW IP anomaly and the E alteration feature that are, as yet, poorly defined (Figure 7-7 below). The well-defined targets, referred to as the eastern, southern, south-western and western anomalies, have been defined using geological mapping, stream and soil sample geochemistry and geophysical surveys using IP with several diamond drillholes in three anomalies (east, south and west) to determine the extent and tenor of mineralisation.

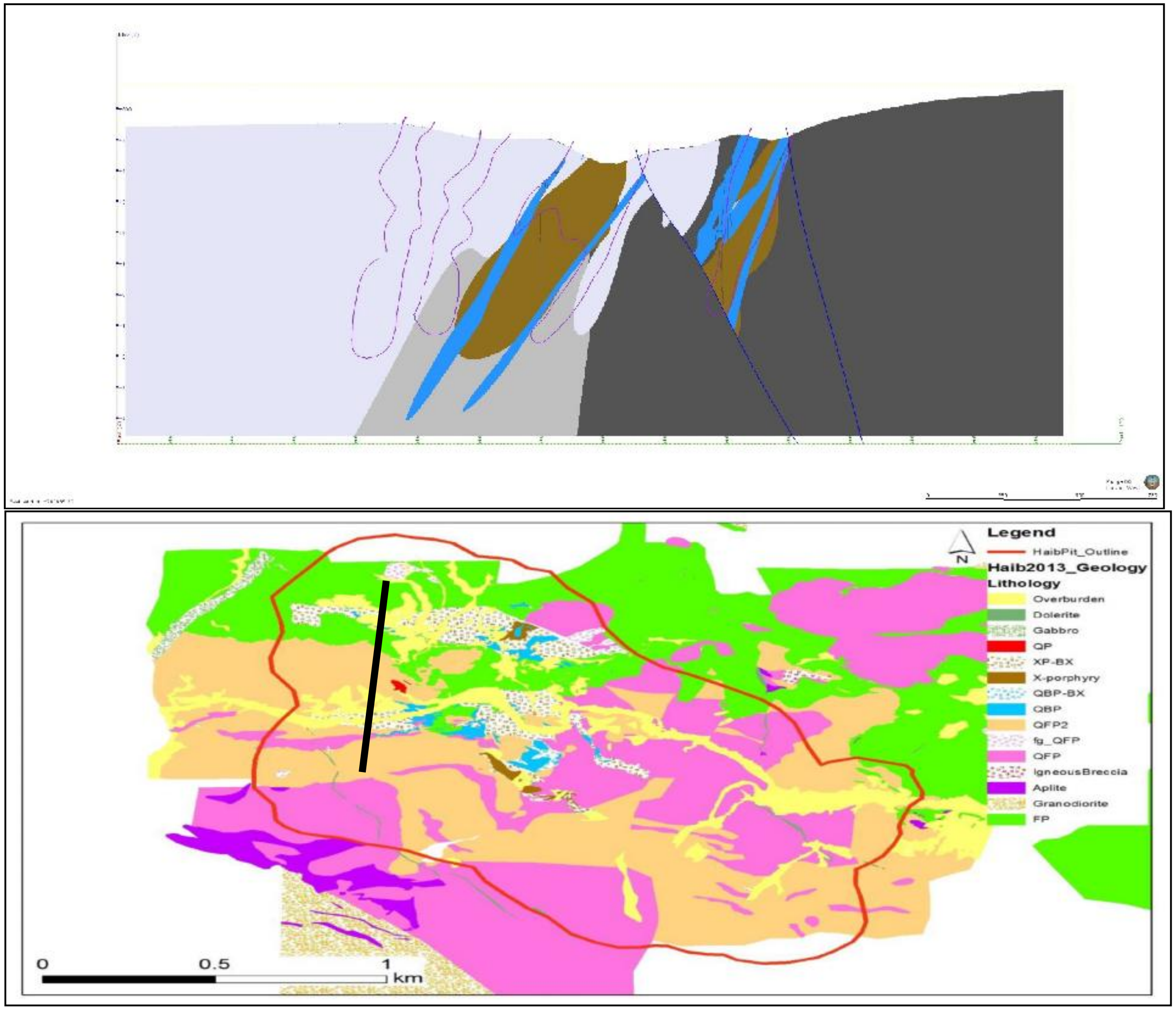


Figure 7-6: North-south cross-section across the western end of Haib, showing steeply south dipping Quartz Breccia Porphyry ("QBP") dykes and hydrothermal breccias (blue and brown) intruding the country rocks (QFP, QFP2 and FP), and truncated by a shear zone in the north and fault in the centre. Potassic Early Dark Micaceous veins are developed mainly in the wall rocks to breccias, particularly to the south. (Teck 2015 [46])

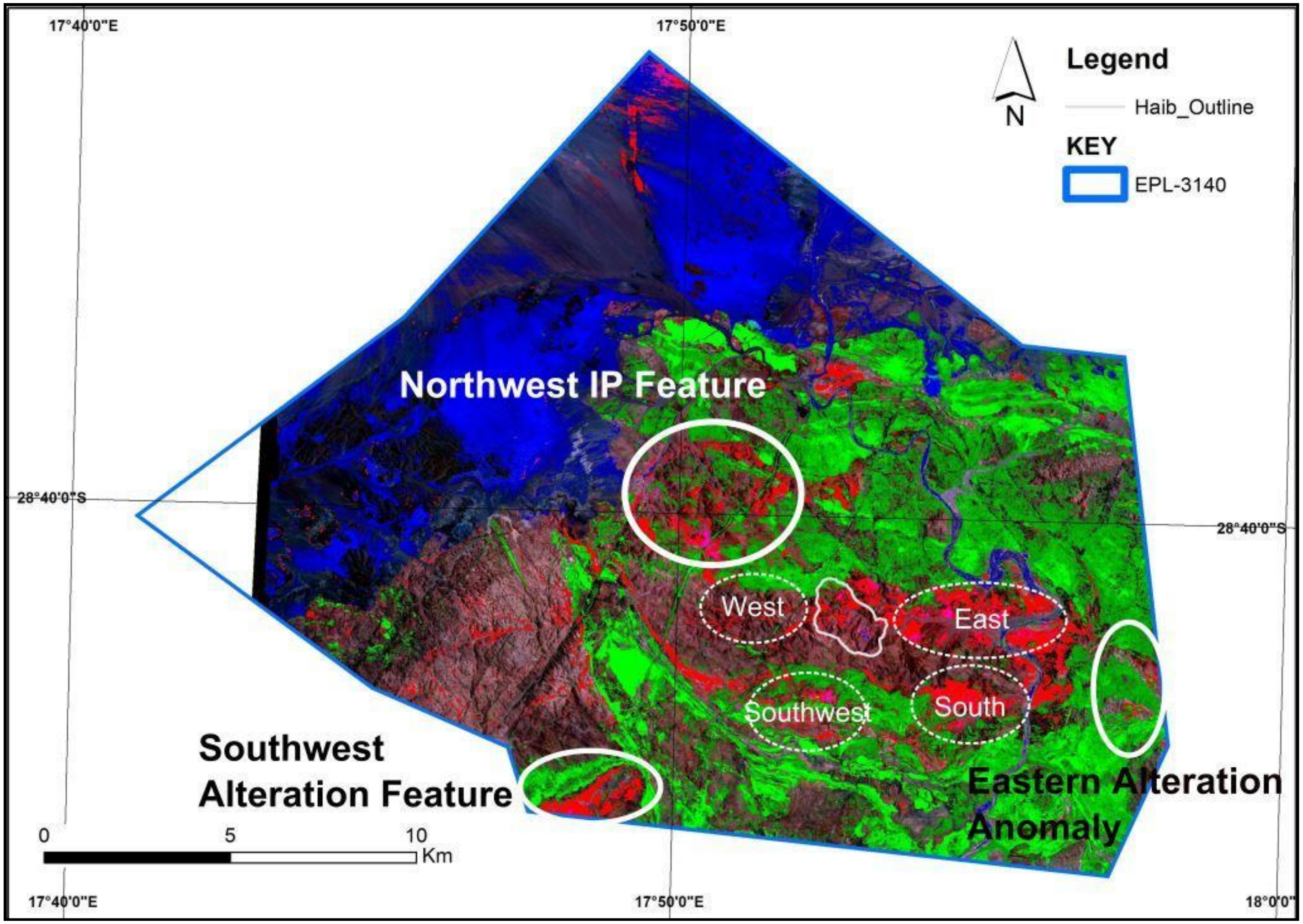


Figure 7-7: Location Map of Exploration Targets: This map shows the main Haib deposit (outlined in solid white at the centre of the coloured area) and the more important anomalies against Hymap Alteration Imagery: - red colours generally correspond to sericite, green to chlorite/volcanics, reddish-brown to Vioolsdrift granodiorite, and blue to Karoo and recent sand cover. (Teck 2015[(47)]).

7.3 The Haib Mineralisation Model [7, 19, 45, 46]

The Haib deposit comprises a large volume of rock containing low-grade copper mineralisation with some accessory molybdenite. At surface, the copper grade varies between three higher grade core zones progressively reducing in grade outwards towards the margin of the deposit. A similar distribution persists below surface in the 300m - 400m levels explored by RTZ, although the recent deep inclined drilling by Teck to 800m depth suggests that the higher-grade zones of mineralisation exist below 400m depth. The surface area in which mineralisation has been identified equates to a surface dimension of about 2,200 x 1,250 metres. The deposit is still partially open to the west (at surface), to the south, and also at depth.

Mineralisation is spatially associated with syn-mineral porphyritic dykes (the QBP) and associated hydrothermal breccias, but there is also considerable vein and disseminated mineralisation in the QFP and FP wall rocks. Molybdenum bearing quartz veins cut both breccia and wall rocks in the high-grade zones. Teck, using very detailed geological mapping techniques, have identified a fore-arc system that strikes E-W, with late-stage oblique fracturing striking N60^{0}W and dipping 70^0 towards the south-west. These fractures are utilized by numerous generations of quartz and Early Dark Micaceous ("EDM") veins accompanied by biotite flooding and increased grades of copper mineralisation. The EDM veins are fractures along which the earliest hydrothermal fluids flowed and are mainly composed of macroscopic biotite-kspar-chalcopyrite ± pyrite, and are typical of porphyry deposits. The sets of EDM veins are parallel to the breccias and dykes and contribute to a high-grade section plunging 30^0 to 40^0 to the east-south-east.

The principal sulphides within the Haib body are pyrite and chalcopyrite with minor molybdenite. Bornite, digenite, chalcocite and covellite are also present locally. There is sulphide zonation where a deep bornite-chalcopyrite assemblage grades outward and upwards to chalcopyrite-pyrite, with a low-grade pyrite ± chalcopyrite fringing zone. There is no major development of a supergene zone, probably due to the high rates of erosion associated with the lower Orange River canyons.

Near-surface oxidation has led to the formation of malachite, azurite, chrysocolla, minor cuprite and chalcocite, generally along fracture zones. Oxide copper rarely extends to depths in excess of 30 metres on these fracture zones. While the oxide zone volumetrically represents a fairly minor proportion of the deposit, grades are significantly above average giving the potential for some leachable copper from the oxide material. These portions of the deposit have not been explored in detail.

In addition, there is a variable thickness of transition zone mineralisation generated over large parts of the deposit, between the surface oxide zone and a pure sulphide (un-oxidised) zone of some 10-20 metres thickness (i.e. completely fresh rock is encountered 30-40m below surface).

Sulphide minerals are disseminated within the rock mass and are also found concentrated in blebs and along veinlets and fractures. Significant mineralisation commonly occurs along quartz veins and in EDM veins.

Gold, silver and molybdenum are trace constituents associated with the copper mineralisation. Molybdenite is occasionally seen as disseminated flakes and in EDM veins, but the majority is hosted within a distinct generation of quartz-molybdenite veins. Assaying for gold, silver and molybdenum was not routinely conducted on drill samples by RTZ but was done later by NCJV on composite samples of their core and some of the RTZ core in their preparatory studies for metallurgical testing, giving an approximate indication of the likely values. HM routinely included determinations of gold, silver and molybdenum in their core drilling and have also re-assayed mineralised sections of 14 RTZ drillholes for these elements (see discussion on this programme under Section 10 below) with results being similar to the NCJV values quoted above.

Outside of the main Haib deposit HM have outlined three satellite targets (see Figure 7-7 above) called the East, West, and South anomalies which have been drilled and evaluated (see Section 9.4 below) as well as four further anomalies – the Southwest alteration feature and Southwest sericite anomaly, the Eastern alteration feature and the North-west IP feature which still require further exploration.

8. DEPOSIT TYPE

The Haib copper deposit is a porphyry copper deposit of palaeo-Proterozoic age [19, 47, 52]. Porphyry copper deposits are a major world source of copper (also molybdenum, silver and gold) with the best-known examples being concentrated around the Pacific Rim, in North America, South America, and areas such as the Philippines. Most of these deposits are relatively young, of Tertiary or Cretaceous age. The United States Geological Survey [52] defines a porphyry copper deposit as follows –

- "One wherein copper-bearing sulphides are localized in a network of fracture-controlled stockwork veinlets and as disseminated grains in the adjacent altered rock matrix;
- Alteration and ore mineralization at 1–4-km depth is genetically related to magma reservoirs emplaced into the shallow crust (6–8+ km), predominantly intermediate to silicic in composition, in magmatic arcs above subduction zones;
- Intrusive rock complexes that are emplaced immediately before porphyry deposit formation and that host the deposits are predominantly in the form of upright-vertical cylindrical stocks and(or) complexes of dikes;
- Zones of phyllic-argillic and marginal propylitic alteration overlap or surround a potassic alteration assemblage; and,
- Copper may also be introduced during overprinting phyllic-argillic alteration events"

The Haib deposit has all of the above defined geological characteristics (see Section 7 above) and is therefore a porphyry copper deposit, being formed within intrusive Proterozoic rocks at 1 880my BP. Porphyry copper systems usually occur along subducted zones and commonly occur in clusters. It is interesting to note therefore, that:- The Lorelei Deposit, some 120km WNW of the Haib (Figure 7-4), is another low grade copper-molybdenum porphyry showing similar alteration zonation and is of a similar age to the Haib [38, 52] and the Tatasberg deposit, some 80 km WNW of the Haib across the border in South Africa (Figure 7-4) is reportedly also a porphyry style Cu-Mo deposit showing typical alteration zoning but is reported to be only some 540my old, although the source of this dating is not reliable; the deposit was explored between 1974 & 1976 by African Selection Trust Exploration ("ASTE") and some 9 diamond drillholes were completed with the best intersection yielding 6% Cu and 32% Mo over a 1m interval but the general average is reportedly some 0.2% Cu [48]. Unfortunately, the detailed reports of ASTE's exploration could not be obtained from the S.A. Geological Survey as they are apparently "lost" in their library.

9. EXPLORATION

In April 2004 Deep South Mining (Pty) Ltd acquired the Haib area under EPL 3140. Subsequently, ownership of EPL 3140 was transferred to HM and Teck, as discussed previously in the Background Sections of this Report, optioned the property and assumed management of the exploration programme.

9.1 The Teck Exploration Programme

Teck took a more regional view of the project than previous operators and did not only focus on the work completed by the NCJV. Their exploration objective was to provide the required data to show that the deposit had potential for large-scale mining, particularly if the tonnage or grade, or both, could be improved and that early stage mining could exploit sufficient high-grade mineralisation to improve the economics of mining. They started a new exploration programme both to investigate the open-ended parts of the deposit (deep drilling and extension drilling) and to explore for new, undiscovered outlying mineralisation. This had not been previously attempted.

Teck, following this model, from 2008 to date, have completed the following work: -

- A regional stream sediment sampling programme collected 276 samples aiming to sample all first and second-order streams every 300m-500m over an area of 320sq.km. This was conducted in 2008 over outlying areas of alteration around the existing Haib deposit. This led to the discovery of four adjacent anomalous zones spaced some 2km from the main Haib mineralisation and it is these anomalous zones that have been geophysically investigated as discussed in later sections. Three of these zones (shown in Figure 7-7 above) have recently been evaluated by diamond drilling and found to be of low grade and caused by distal veining from some unknown porphyry intrusive.

- A total of 32 diamond drillholes (totalling 14,252 metres). These were drilled within the historically defined main mineralisation and on the Eastern, Southern and Western IP / soil geochemical anomalies (discussed in Sections 9.4 & 10.2 below).

- Using the Anaconda mapping method, which maps in detail the lithology, alteration, vein type, orientation and intensity on separate overlays, they have mapped about 75% (205 ha) of the area around the 275 ha. main deposit (at a scale of 1:10,000) and all (90 ha.) of the main deposit at 1: 2,000 scales; Teck have also mapped the Eastern and Southern IP defined anomalies at 1: 10,000 scales, while the vein zone at Haib West has been mapped at 1: 2,000 scales (Locations are shown in Figure 7-7 above).

- They have re-logged all of the available (108 out of 120) old RTZ drillholes in detail, again using the Anaconda method. These were all located within the Main Haib deposit.

- They have re-sampled 14 of the old RTZ drillholes to compare the assay results obtained by RTZ for copper and also to determine the grade of gold, silver and molybdenum (Figure 10-2 below).

- They completed some 83-line kms. of pole-dipole Reconnaissance Induced Polarization (RIP); and another 6-line kms. of Audio Magnetotellurics (AMT).

  (Note: - AMT is a high-frequency magneto-telluric technique for shallower investigations. While AMT has less depth penetration than MT, AMT measurements often take only about one hour to perform, although deep AMT measurements during low-signal strength periods may take up to 24 hours, and use smaller and lighter magnetic sensors.)

- They collected 636 soil samples on grid lines 150m apart with sample spacing of 50m covering an area of 400 hectare across three of the satellite targets – the South, Southwest and West anomalies (Location - Figure 7-7 above).

- They constructed a 3-D geological model of the Main Haib zone using Leapfrog geo-modeling software (see Figure 9-1 below). This model combines all the surface and down hole geology, assays and geochemistry to constrain the grade envelope for a resource estimate.

9.2 Teck's Geophysics

Various geophysical techniques have been applied over the Haib deposit on several occasions. The earliest documented geophysics for which records exist was an Electromagnetic survey (EM) conducted by J. Shepherd of Falconbridge in March 1964.

A further significantly more detailed IP and resistivity survey was conducted by RTZ in December/ January 1974-75. This covered the bulk of the main mineralised area.

Teck proceeded to complete RIP, PDP and AMT geophysical surveys initially over the main Haib mineralisation and then extended their surveys to cover targets generated by a study of alteration patterns in the Proterozoic country rocks in EPL 3140. These programmes were conducted in-house. Figure 9-2 below shows the location of RIP sections completed across the main Haib mineralised zone with a 3D representation showing the Haib drilling. The impact of the disseminated sulphides in the main Haib body is well represented by the zones of red and pink (high chargeability).

The fairly extensive geophysical survey programme over alteration anomalies around the main Haib mineralised body determined several additional zones of high chargeability. These geophysical anomalies, together with detailed geological mapping to show alteration and geochemical soil, stream and rock chip sampling results allowed Teck to prioritise follow-up evaluation programmes of these anomalies which are on-going (for their location see Figure 7-7 above).

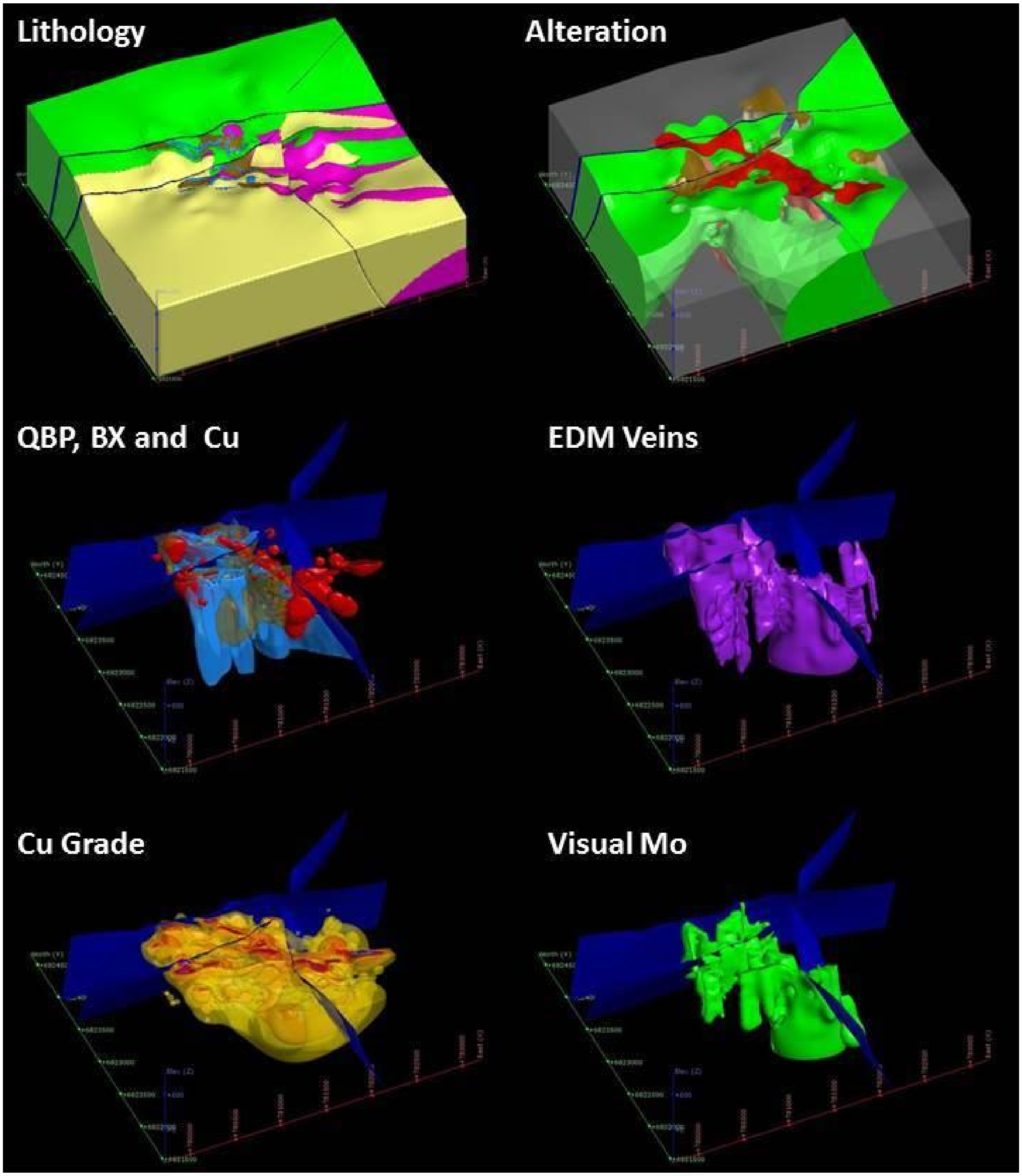


Figure 9-1: Three Dimensional Models: A compilation of 3-D models of the Main Haib deposit (Source: Teck, 2015)

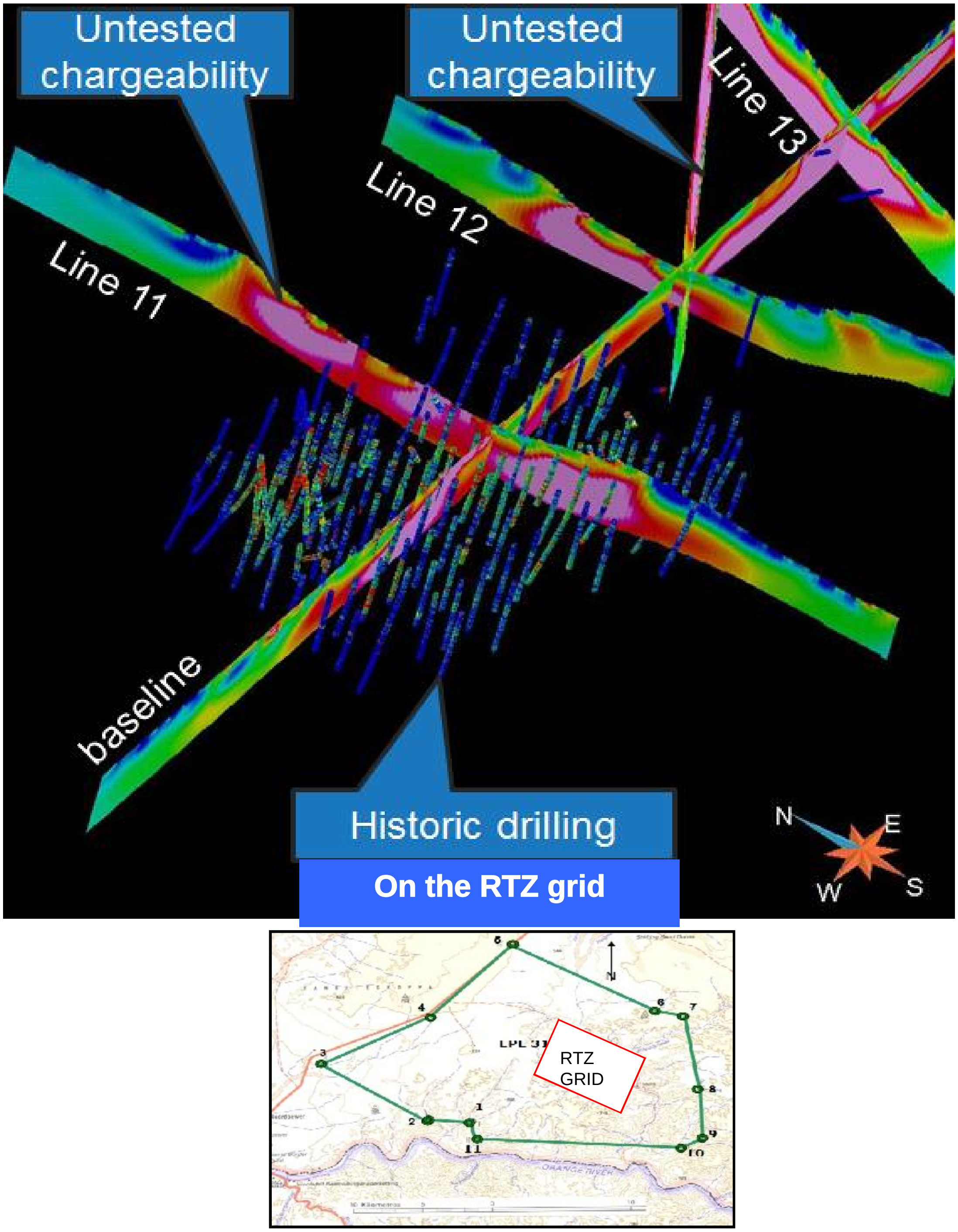


Figure 9-2: A 3-D diagrammatic representation of the Teck IP geophysical section lines across the main Haib deposit. The pink and red zones adjacent to the drillholes show the zone of mineralisation with a high chargeability. (Source Teck 2012)

9.3 Teck's Geochemical Surveys

9.3.1 Stream Sampling

This sampling campaign was aimed at evaluating the outlying areas of the licence and none of the main Haib mineralisation area was sampled by Teck. The sampling was completed in 2008 and all the large third-order streams were avoided since these would be much diluted. First and second-order streams were sampled every 300m to 500m by collecting roughly 1-2Kg of sample from trap sites using a stainless-steel shovel, dry sieving these to -2mm and further to -80# size using stainless steel sieves (brushing the sieves between samples and washing them every day) and packaging these in a brown paper sample bag with a sample number tag inside and outside of the bag. The GPS location was taken and recorded. Every 20th sample is duplicated by taking another sample within 1 or 2 metres of the first site. Standard and Blank samples are inserted later on a 1:20 frequency but randomly inserted in the sampling sequence.

9.3.2 Soil Sampling

Since RTZ soil sampling coverage around the main Haib body was quite extensive, Teck have extended their grid lines into the outer regions of the licence using the same orientation. The samples were collected on 150m line spacing using 50m sample spacing. This campaign has collected 636 samples over an area of 400 hectare. The procedure employed is to dig a hole to a depth of some 10cm. using a stainless-steel shovel, dry sieving these to -2mm and then -80# size using stainless steel sieves (brushing the sieves between samples and washing them every day) and packaging these in a brown paper sample bag with a sample number tag inside and outside of the bag. The GPS location is taken and recorded. Every 20th sample is duplicated by taking another sample within 1 or 2 metres of the first site. Standard and Blank samples are inserted later on a 1:20 frequency but randomly inserted in the sampling sequence.

9.4 Teck's Geological Mapping

Teck use a geological mapping method which results in at least 3-overlays for mapping of structural, lithological and alteration features. The more detailed mapping in and around highly mineralised areas add another “vein” overlay to this map. The various features are colour coded.

9.5 Teck's Other Targets

Outside of the main Haib deposit Teck outlined three satellite targets, as indicated on Figure 9 above. The eastern anomaly, with extensive sericite alteration zones, high molybdenum geochemical results and a defined IP anomaly, has been evaluated by four vertical diamond drillholes with a total depth of 1,525.35m (see Table 8 below) with only minor traces of mineralisation.

The southern anomaly (Figures 7-7 above and 9-3, 9-4 below) is also well defined by extensive sericite alteration, some copper staining with haematite/limonite and gypsum associated with quartz vein sets, so-called D-veins in porphyry system terminology (photograph 5 below) and a distinctive IP response. On surface this anomaly extends over 1.2km along strike and 350m – 500m across strike; it appears to be steeply dipping to the south. Four diamond drillholes totalling 1,484.34m of which 3 holes were angled to the north and one to the south were used to evaluate this anomaly, (see Figure 9-5 below) but assay results indicated less than 0.2% Cu in zones where there is a high density of D-veins and <0.1% Cu elsewhere. The strong IP anomaly is probably the result of the abundant pyrite in the veins. This vein zone has been interpreted as being distal from a further porphyry system but because of thrusting and late-stage normal faulting, the location and depth of this body are difficult to estimate. The drilling clearly defines a lower contact for the vein zone.

Photograph 5: Weathered outcrop showing iron-stained (after pyrite) quartz D-Veins at the Haib South Anomaly.

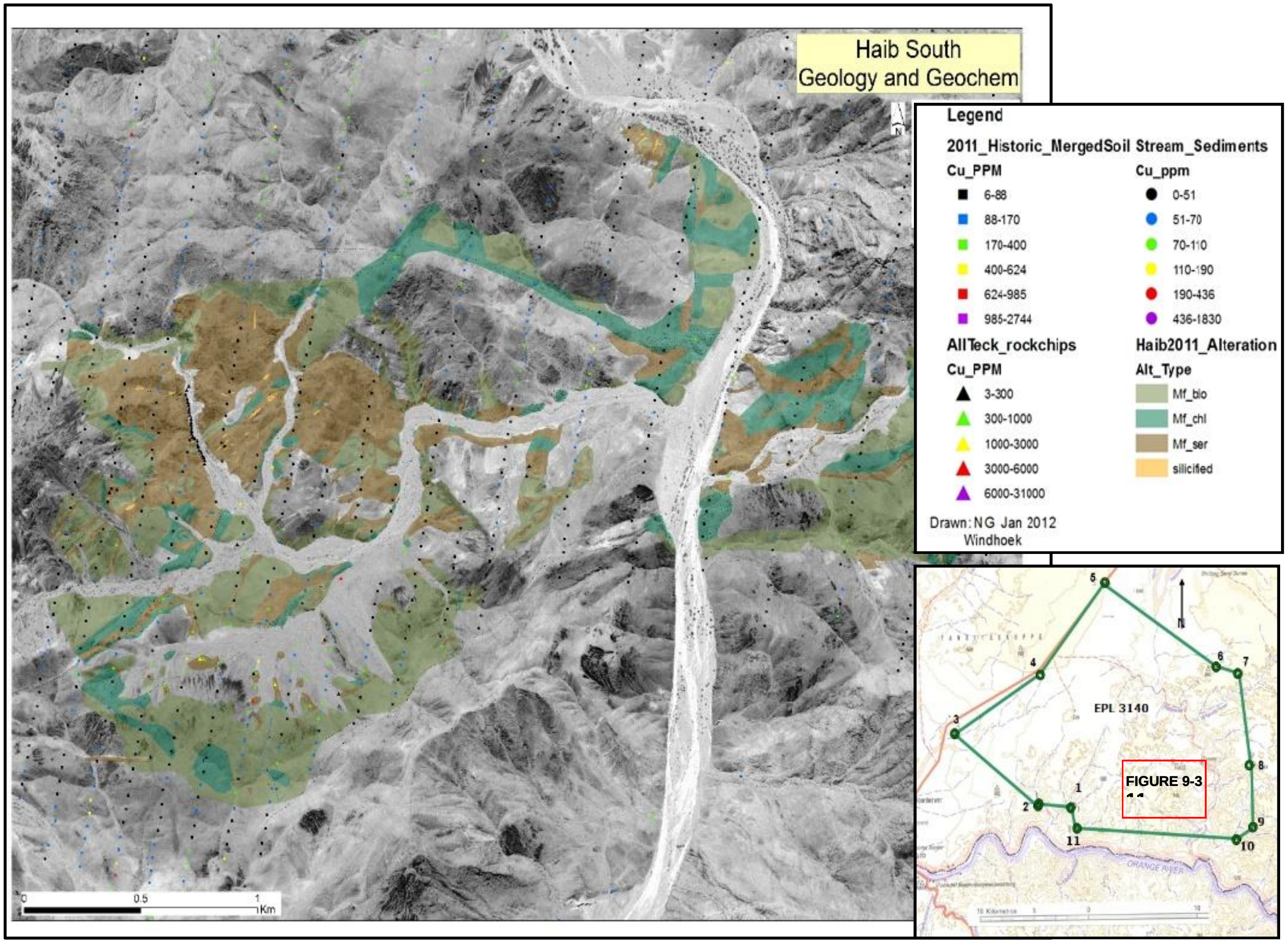


Figure 9-3: Alteration Geology: Map showing the alteration geology and geochemical sampling of the Haib South anomaly (Source Teck 2012[42])

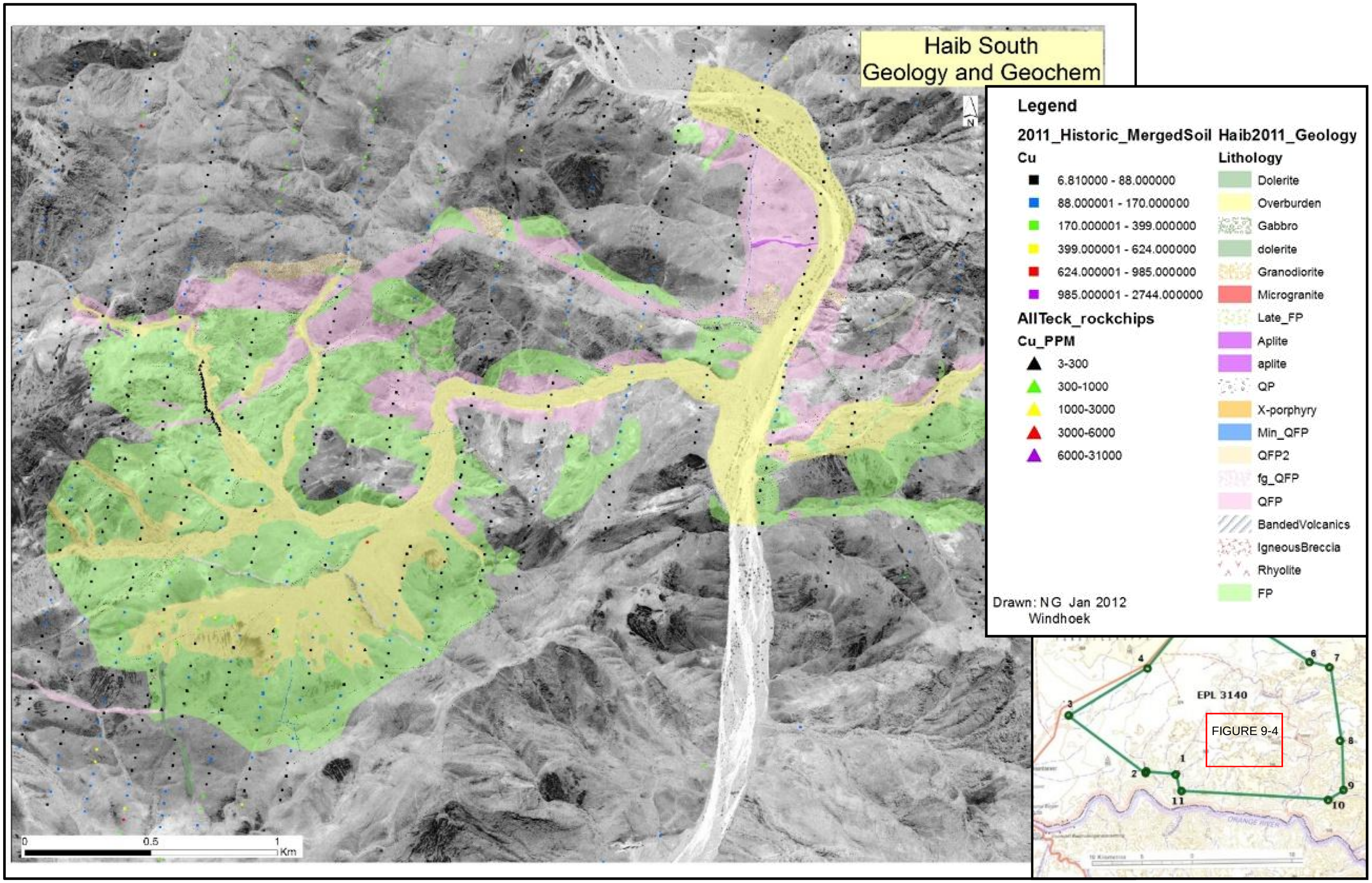


Figure 9-4: Map showing the lithology of the Haib South anomaly (Source Teck 2012[42])

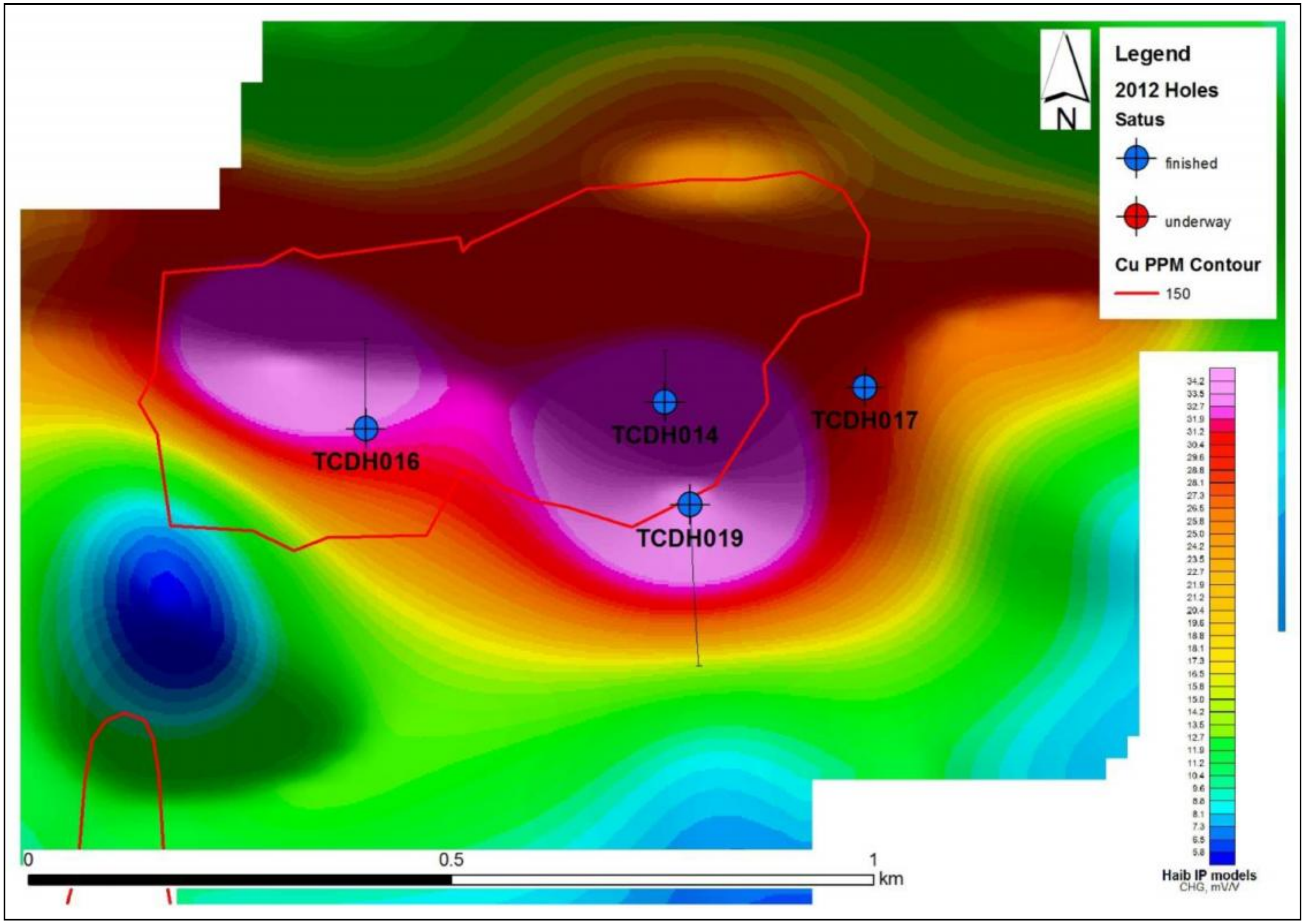


Figure 9-5: Map showing the location of drillhole collars overlaid on the IP Chargeability contours (50m depth slice of pole-dipole lines) & the Cu in soil >150ppm contour line – Haib South Anomaly (Source Teck 2012[42]).

The Western anomaly (Location - see Figure 7-7 above) consists of a km-scale soil anomaly coincident with an 800m long, NE trending and SE dipping quartz vein zone (photograph 6 below) truncated in the Northeast by a shear zone. The veins are predominantly A- and B-type quartz veins with only minor EDM veins, and are analogous to those found at the main Haib deposit. The Western anomaly was drilled by RTZ using a single vertical hole. Teck have now drilled two inclined diamond drillholes totalling 735.37m but these were completed before the detailed mapping programme and may not have been sited optimally (see Figures 9-6, 9-7, and 9-8 below). The one Teck hole intersected 44m of 0.22% Cu with a high-grade section of 4m at 0.4% Cu. Again, this anomaly has been interpreted as a distal portion of a separate porphyritic intrusive.

There is some alteration and IP evidence which outlines a further four targets (see Figure 7-7 above) which will require follow-up geophysical, geological mapping and geochemistry work to confirm their potential as exploration targets for drill investigation: -

- The North-western IP anomaly has poorly defined soil geochemistry and has not yet been drilled by HM.
- The South-western anomaly contains extensive sericitic alteration with zones of pyrophyllite and alunite indicating it is very high in the porphyry intrusive alteration system but with no indications of near surface mineralisation.
- The Eastern alteration feature consists of a quartz-rich sericitic alteration zone with minor copper staining within the FP to the immediate south of a dyke of QFP dipping steeply (75 – 80^0) to the Southwest.

Photograph 6:

Example of the vein zone at Haib West, with each horizontal fracture representing a quartz vein which weathering has broken open.

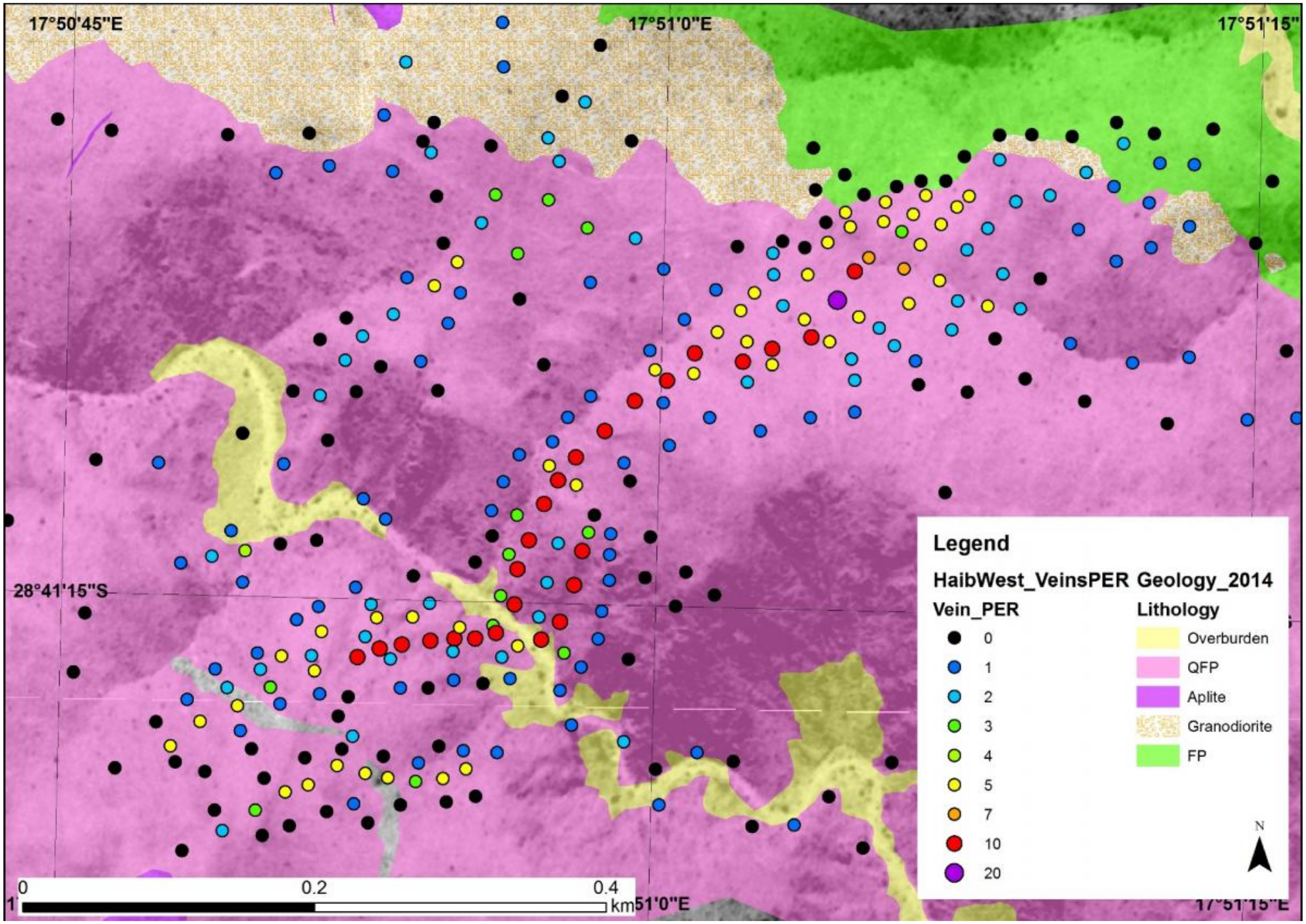


Figure 9-6: Map showing the geology and vein densities (%) at Haib West (Source Teck 2015[45])

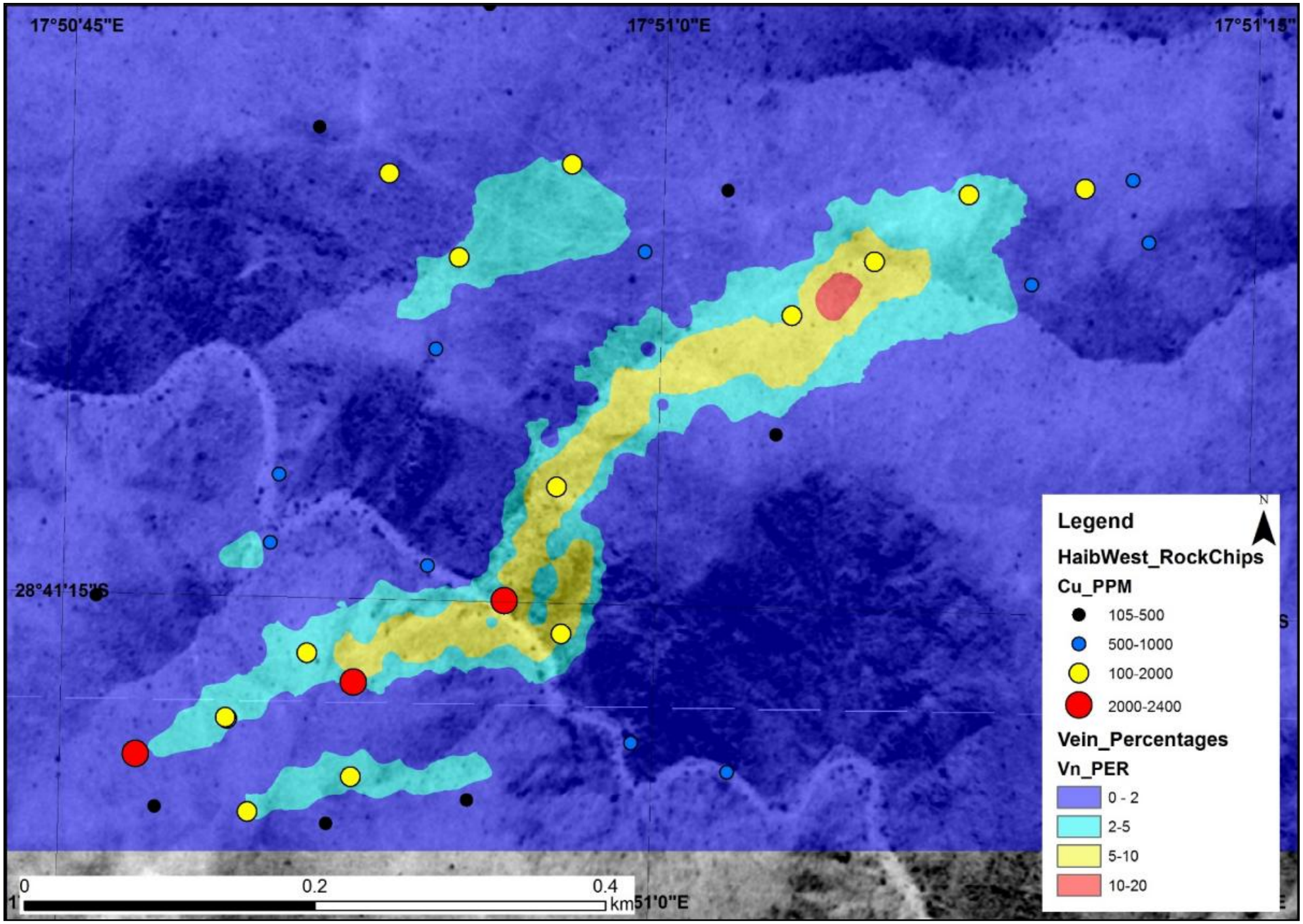


Figure 9-7: Rock Chip samples collected across the vein zone identified at Haib West, with a background image of contoured vein percentages estimated from outcrop locations in Figure 9-8 below (Source Teck 2015[46]).

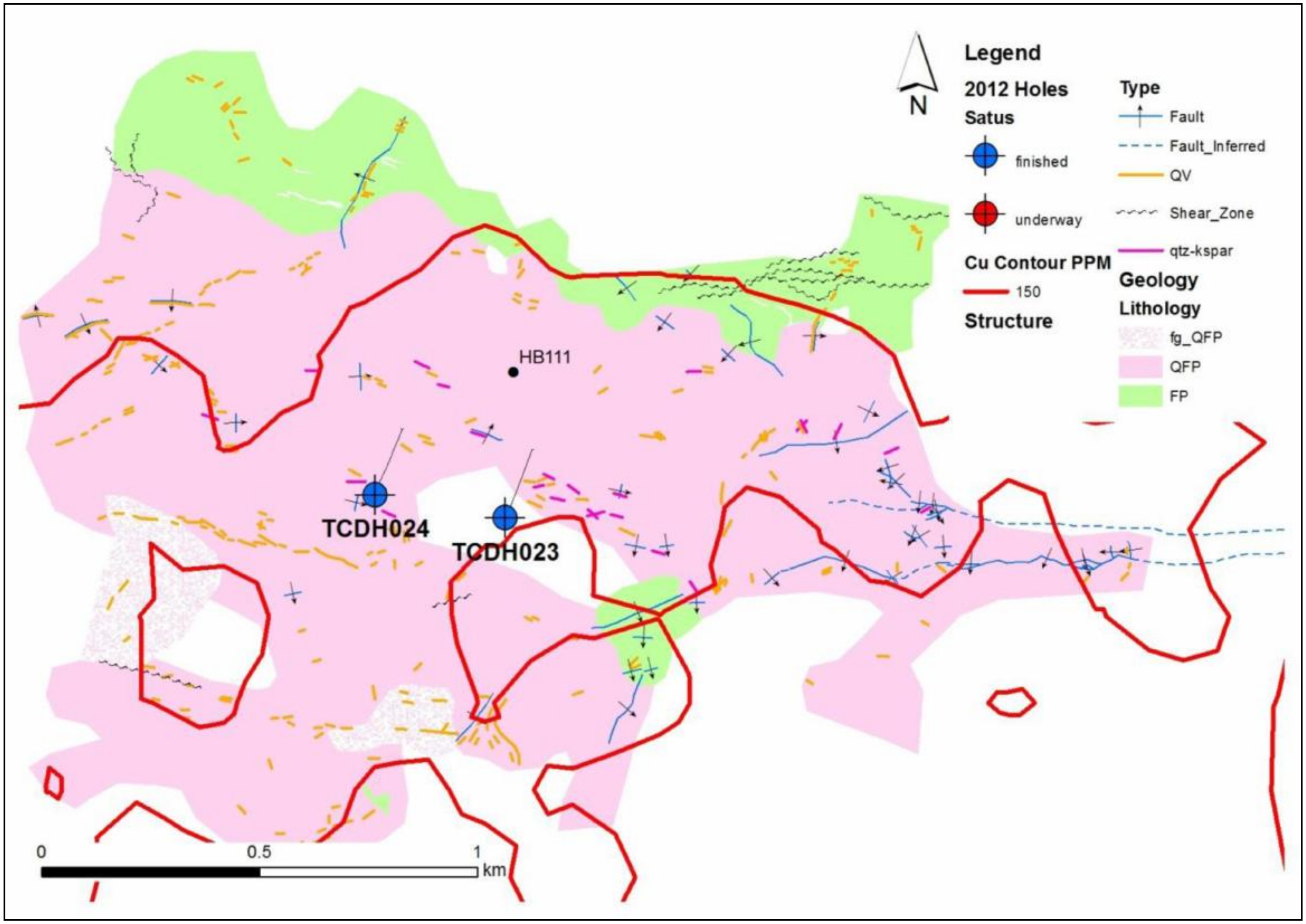


Figure 9-8: Drillhole Locations – Haib West: Plan map of Haib West showing the position of the holes drilled in 2012, targeting a zone of veining in the centre of a broad soil anomaly (Source Teck 2013[45])

10. DRILLING

10.1. Historical [7]

At least five separate drilling programmes have been conducted at the Haib; for dates of these programmes see the History Section above.

The first drilling was completed by Falconbridge who drilled eleven drillholes into the deposit in three principal areas of interest. Total drilling of some 1,012 metres was completed. The average grade of the drillhole intersections was given as 0.33% Cu. Very little of this data remains other than the drill core assays and their location in the field. It is not really possible to comment on this programme.

After Falconbridge, King Resources conducted a drilling programme of 21 holes totalling 3,485 metres. Again, this programme has very little useful data surviving, although drill assays are available and the drillhole collars have been located.

Most of these earlier holes were blocked or difficult to locate.

Subsequently, RTZ completed one hundred and twenty diamond drillholes, mostly vertical, on a systematic 150 metre square grid giving a total of 45,903 metres drilled (Figure 18). Holes were on average 300-400 metres deep. These cores are preserved in a shed (see photographs 3 and 4 above) at the old RTZ campsite and are available to study although some mineralised sections are reduced to quarter-core by assay and re-assay campaigns. The information from these drillholes was verified by GFM and incorporated into their geological model. This information was therefore used by Behre Dolbear in the Haib model evaluation presented in section 6.3.4. of this report.

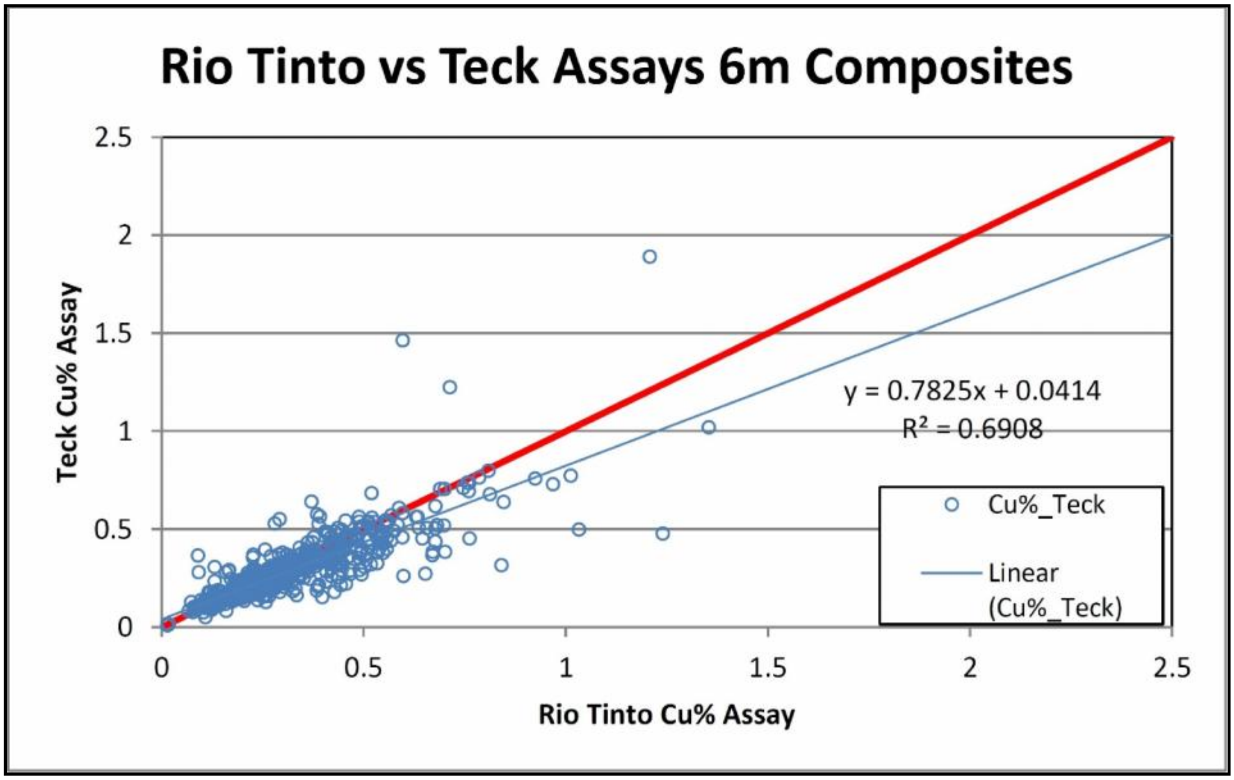
All drillhole assay data is based on diamond drill core, generally “N” or “B” sizes. Drillhole spacing was generally on a regional 150 metre square grid. The RTZ drillholes are mostly vertical, while the earlier Falconbridge and King Resources drillholes are inclined. One section line, 865_{00} E was partially drilled by RTZ at 25 metres spacing across the zone of high-grade mineralisation. This was the line along which the adit was developed by the NCJV (photograph 2 above).

Sample recovery was reported to be generally good. Most of the historical drillholes were hammer-split and half core composites were sent for assay. The RTZ cores were sampled over 2 metre intervals for determination of total copper and, where appropriate, acid soluble (oxide) copper. Composite samples from each drillhole were tested metallurgically to determine recoverable copper and were assayed for molybdenum, silver and gold indicating average contents of 25 g/t Mo, 0.01 g/t Au, and 0.9 g/t Ag. The reliability of these numbers cannot be assured as assay certificates are not available.

From all of this information Venmyn Rand captured an electronic database of the available 1963-1975 drillhole data using drillhole logs as the original assay data sheets were unavailable. The database comprised 152 drillholes – 120 from RTZ, 21 from King Resources and 11 from Falconbridge.

To this database have now been added the 13 holes drilled by GFM and the 32 drillholes completed by Teck.

In 2010 and 2011, Teck quartered 3,714 metres of RTZ core from 14 drillholes (Figure 10-1 below) on a composited 3-metre sample interval and submitted them for re-assay using an Aqua Regia digestion method and an Inductively Coupled Plasma Emission Spectrometry (ICP-ES) technique to provide a 24 element determination; the RTZ composite samples were done on a 2-m sample interval whereas Teck composited at a 3-m interval – this means that a comparison of average elemental values can only be made at 6m intervals. The 619 x 6-m average value comparison for copper revealed that below 0.6%Cu the assay results are statistically identical but that bias (~15% positive bias in the RTZ data) creeps into the data above the 0.6% Cu level as shown graphically in the binary X-Y plot below.



It is probable that the ~15% positive bias in the RTZ >0.6% Cu results may be due to RTZ using a 4-acid digestion method which would release copper from silicate minerals, for example the copper in the biotite lattice in the high-grade zones where EDM veins are ubiquitous.

It should be noted that some of Teck's check assays of RTZ core completed in 2010 also used a fire assay in addition to the ICP-ES method, but since virtually all values returned <5ppb Au, it was decided to discontinue the fire assay as a routine assay method.

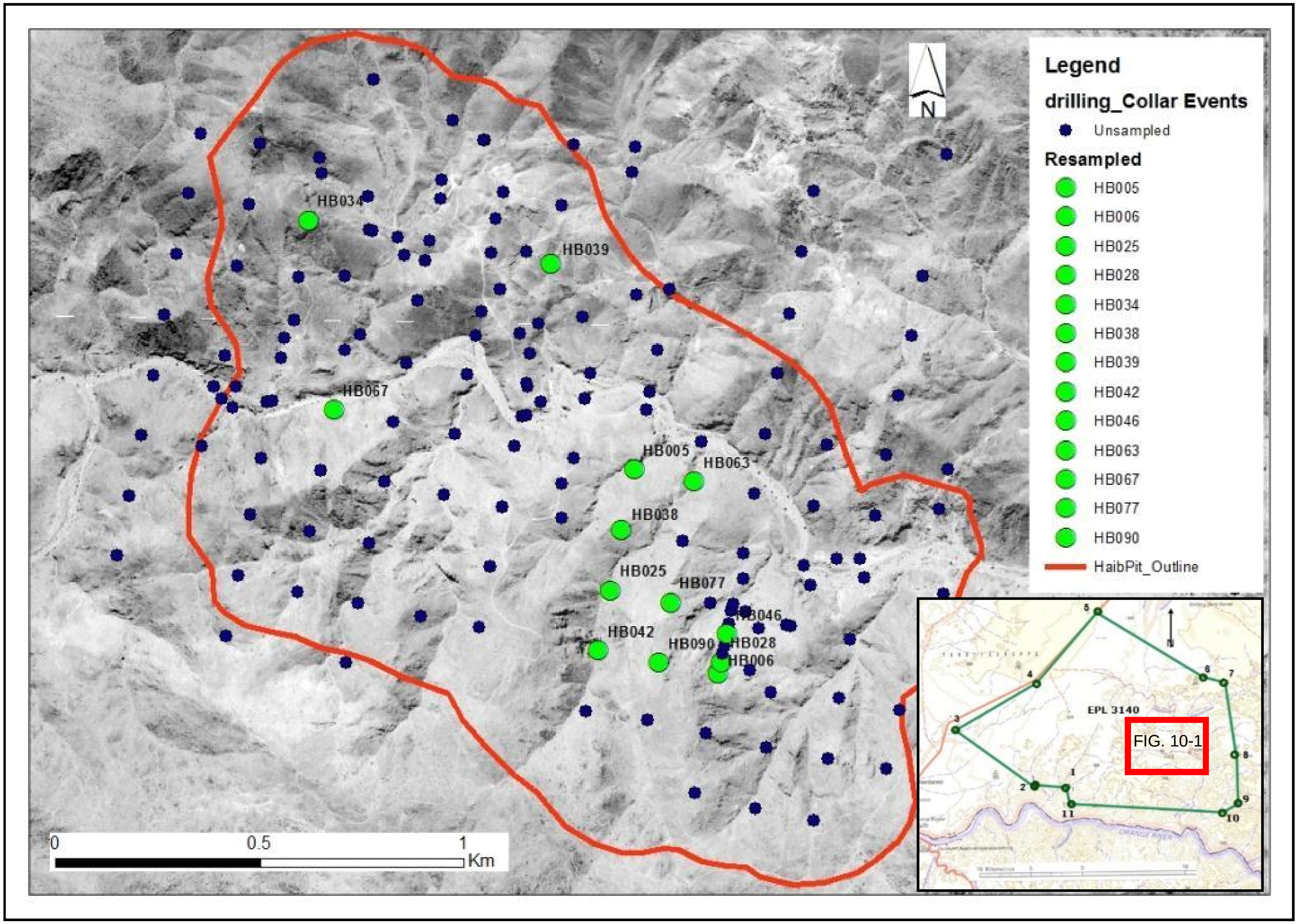


Figure 10-1: Location of RTZ Drillholes: Plan showing location of RTZ drill collars and those re-assayed by Teck (Source: Teck 2012[42])

An attempt was made by the NCJV to locate and resurvey all drillhole collars completed during the Falconbridge, King Resources and RTZ programmes. The data from this survey is available although not included in this report. Where existing data was available, the eastings and northings were generally found accurate but there were significant discrepancies (up to 80 metres) in the reported drillhole elevations. This factor represented a constraint on the accuracy of the data for geological modelling and on the confidence limits placed on the mineral estimates, but it was not considered that it would have a significant impact on the overall estimate figure as discussed later.

This issue was subsequently resolved by the NCJV which commissioned an Orthophoto survey of the area and generated a new surface topographic plan.

The NCJV/ GFM core drilling programme completed a further 12 infill drillholes for analytical purposes and another 5 large-diameter drillholes for geotechnical work. Technical data is available for these holes. These were reported in an October 2004 report titled "Independent Technical Review, the Haib Copper Porphyry Project, Namibia" [7].

10.2 Teck's Drilling

The most recent drilling programme at Haib was completed by Teck between 2010 and 2014 and comprised 32 diamond drillholes totalling 14,252m. Figure 10-2 below shows the location of the Main Haib deposit drillholes (including historic drillholes) and Photograph 7 shows a diamond drill-rig on site.

These drillholes were used to evaluate several target zones; the first group of 22 holes totalling 10,507.92m was drilled within the existing main Haib mineralised body; the holes were drilled to test: -

- the predictability of the mineralisation grades in the model derived from historical assay data,
- the higher-grade portion of the mineralised body and,
- the deeper portions of the known mineralisation with the deepest hole at 806m depth (some 800m below surface).

Photograph 7:
This photograph shows one of the larger drill-rigs used by Teck to drill the deeper drillholes at the Haib eastern anomaly area. This rig is situated in the Volstruisrivier at Haib.

Table 10-2 below gives the basic data for all of these 32 drillholes with the Copper and Molybdenum results for significant intervals.

The second group of diamond drillholes tested for mineralisation at the Eastern, Southern and Western combined soil and geophysical anomalies and consisted of 10 holes totalling 3,745.06m. The location of these anomalies is shown in Figure 7-7 above.

10.2.1 Teck's Protocol for Drillhole Surveys

The drillhole collar locations are surveyed using a hand-held GPS at the start of the drillhole and a certified land surveyor using a differential GPS surveys all of the drill collars at the end of the programme. Down-hole surveys using a Reflex EZ-Com multi-shot tool are performed on holes in the main Haib body at 6m intervals as rods are pulled from completed holes. Down-hole surveys of the exploration holes into peripheral anomalies use a Reflex EZ-Com single shot tool at 100m intervals during the course of drilling. These instruments have a stated accuracy of 0.1 degrees of dip and azimuth.

10.2.2 Teck's Procedures for Drill Logging and Sampling

All drillhole cores are collected daily and stored in a galvanized steel tray at the core yard. The cores are washed to remove all residual cuttings and drill additives. The core is then measured to determine core recovery and Rock Quality Designation ("RQD"). The average recoveries reported by RTZ were >95% and Teck have measured average recoveries of >99% in the main Haib body and averages of >98% in the more altered peripheral Anomalies. The whole cores are then photographed.

The core is logged for lithology, alteration, structural elements, and mineralisation before being marked up for core cutting and sampling, the core sample length being at the discretion of the geologist, bearing in mind the wish to constrain well mineralised intervals and lithological breaks with recommended minimum 1m and maximum 3m length; to date the sampling tends to average 2m in length. The entire hole is sampled.

The core is halved sampling one half only, although early drillhole core from 2010 and 2011 used quarter-core duplicates, subsequent drillhole sampling has used half-core duplicates. Core cutting is done on site using a water-cooled diamond saw with the cutter being assigned one hole only and prohibited from wearing any jewellery; The saw is cleaned twice daily using a concrete brick and simultaneously the coolant water settling tanks (2 sequential tanks per machine) are also emptied and cleaned. The half core sample is bagged in good quality plastic sample bags with one sample number tag inside and a duplicate number tag attached to the outside of the bag. The sample bags are batched and transported by Teck personnel to Analytical Laboratory Services, an independent commercial laboratory in Windhoek where the samples are crushed, milled, and split with representative splits shipped by Teck in batches sealed in a box using FedEx couriers to an independent commercial laboratory, Acme Analytical Laboratories, now a subsidiary of Bureau Veritas in Vancouver, Canada for assay. Teck core samples are batched with a blank, standard and duplicate sample inserted every 20 samples. The Windhoek laboratory duplicates every 20th crushed sample to check for any bias after splitting of the crushed sample and for combined preparation and analytical variation (see discussion under section 11 below).

The above protocols ensure minimum probability of sample contamination and the chain of custody is also well defined and ensures minimal opportunity for third party tampering with samples.

10.2.3 Results

The details of holes drilled, their location and significant intersections is summarized in Table 10.2 below.

Several of Teck's drillholes were drilled deeper than the average RTZ hole (about 400m). The deepest Teck hole was TCDH-06 drilled to 842.78m. The log of this hole is shown in Figure 10-3. The hole produced anomalously high copper results (Table 10-2) as the average grade throughout the drillhole was significantly higher than predicted from the mineralisation model derived from previous RTZ drilling.

Table 10-1: Selected assay results for drillhole TCDH-06.

Hole No.	From (m)	To (m)	Copper (%)	Molybdenum (%)
TCDH-06	0	842.78	0.285	0.011
Including	349	842.78	0.36	0.018
Including	537	658	0.5	0.027

This hole also illustrates the point that historical, vertical drilling may have under-estimated the extent and tenor of the high-grade plunging EDM vein sets.

10.3 Contributing QP's Comment on Core Sample Representivity and Bias

Since both the RTZ and Teck drilling core recoveries were respectively >95% overall, our (Peter Walker's and Dean Richard's) opinion is that core sampling fairly represents the values of the particular intersections reported on and core loss, broken ground or voids do not materially impact on accuracy or reliability of results.

The mineralisation is to a large extent in disseminated form and there are only small differences between the sample length and the true thickness of mineralisation for the majority of the drillholes. However, detailed evaluation of the higher-grade sections of the main Haib body which have additional mineralisation in sheeted veins following fractures dip steeply to the south. Teck have detailed surface mapping and logging of inclined drillhole intersections (both RTZ & Teck) through this zone and are confident that their calculations of true thickness of mineralisation within this zone are accurate. Since vein control of high-grade mineralisation is apparent in some locations, care is exercised in sampling intervals where veins run at a low angle to the core axis, since these intervals may overestimate the grade; these occurrences are rare and in our opinion the assay results fairly represent the true grades with minimal bias.

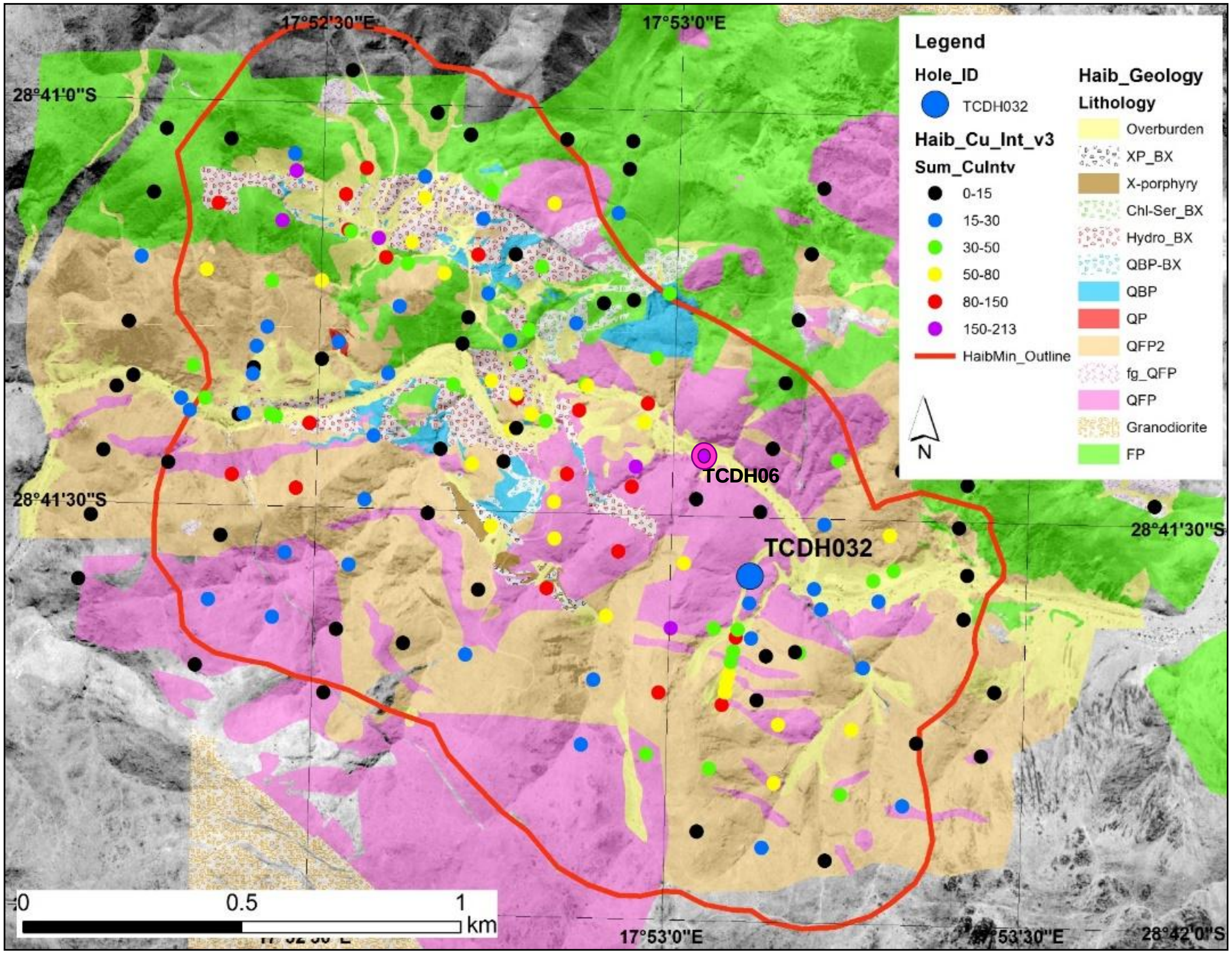


Figure 10-2: Location of All drillholes into the Main Haib Deposit: This map shows the historical drilling together with the recent Teck drilling. The location of drillhole TCDH06 is shown and its log is shown as Figure 10-3. (Source: Teck 2015 [45])

Table 10-2: - Details of the Teck Drilling with Significant Intersections.

Hole ID	Target	X	Y	Z m	Length m	Azimuth	Dip	From (m)	To (m)	Interval (m)	Cu %	Mo %
TCDH-01	Haib East	784102	6823216	386	434.17	360	-90	No significant Intersections				
TCDH-02	Haib East	783201	6823112	348	350.04	360	-90	No significant Intersections				
TCDH-03	Haib East	784698	6823388	309	383.23	360	-90	No significant Intersections				
TCDH-04	Haib East	784709	6822725	328	357.91	360	-90	No significant Intersections				
TCDH-05	Deposit	781662	6822222	556	806.52	14	-80	0	806.52	806.52	0.16	0.005
Incl.								218	245.2	27.2	0.22	0.002
TCDH-06	Deposit	781802	6822906	397	842.78	194	-50	0	842.78	842.78	0.28	0.011
Incl.								349	842.78	493.78	0.36	0.018
Incl.								537	658	121	0.5	0.027
TCDH-07	Deposit	781667	6823026	398	822.86	195.5	-65	0	822.86	822.86	0.25	0.008
Incl.								231	405	174	0.32	0.006
Incl.								704	764.9	60.9	0.38	0.012
TCDH-08	Deposit	781624	6823560	468	370	30	-60	No significant Intersections				
TCDH-09	Deposit	781366	6823047	428	602.11	15	-55	0	602.1	602.1	0.16	0.006
Incl.								63	196	133	0.36	0.010
TCDH010	Deposit	781639	6822881	415	799.11	182	-65	0	799.9	799.9	0.29	0.012
Incl.								172	799.9	627.9	0.31	0.014
Incl.								227	515	288	0.37	0.018
Incl.								269	308	39	0.53	0.020
TCDH011	Deposit	780763	6823093	432	601.06	7	-60	0	601.06	601.06	0.10	0.003
Incl.								170	215	45	0.53	0.002
TCDH012	Deposit	780490	6823091	431	156.26	5	-50	No significant Intersections				
TCDH013	Deposit	782228	6822642	387	600.77	186	-55	0	600.77	600.77	0.19	0.003
Incl.								14	151	137	0.28	0.001
Incl.								86	139	53	0.34	0.001
Incl.								20	29	9	0.63	0.004
TCDH014	Haib South	784364	6819712	380	351.33	360	-80	No significant Intersections				
TCDH015	Deposit	781201	6823323	519	464.02	10	-60	0	464.02	464.02	0.2	0.002
Incl.								188	242	54	0.26	0.002
Incl.								50	82	32	0.56	0.005
TCDH016	Haib South	784010	6819680	435	311.06	360	-70	No significant Intersections				
TCDH017	Haib South	784601	6819729	351	393.85	360	-90	No significant Intersections				

(See continuation of Table 10-2 on next page)

Table 10-2 continued

Hole ID	Target	X	Y	Z m	Length m	Azimuth	Dip	From	To	Interval	Cu	Mo
								(m)	(m)	(m)	%	%
TCDH018	**Deposit**	**781566**	**6823254**	**494**	**460.95**	**14**	**-60**	0	455	455	0.1	0.002
Incl.								63	69	6	**0.26**	**0.023**
Incl.								269	279	10	**0.26**	0.001
TCDH019	**Haib South**	**784394**	**6819590**	**415**	**428.1**	**182**	**-60**	**No significant Intersections**				
TCDH020	**Deposit**	**781600**	**6823460**	**476**	**497.12**	**360**	**-90**	0	497.12	497.12	**0.24**	0.006
Incl.								116	472	356	**0.3**	0.008
Incl.								256	270	14	**0.39**	0.005
TCDH021	**Deposit**	**782397**	**6822631**	**404**	**314.25**	**190**	**-60**	**No significant Intersections**				
TCDH022	**Deposit**	**781265**	**6822888**	**435**	**477.48**	**12**	**-60**	0	477.48	477.48	**0.21**	**0.011**
Incl.								332	425	93	**0.44**	0.007
Incl.								383.65	415.52	31.84	**0.79**	**0.010**
TCDH023	**Haib West**	**778372**	**6823136**	**494**	**376.93**	**21**	**-60**	0	376.83	376.83	0.08	0.001
Incl.								180	204	24	**0.2**	0.001
Incl.								276	296	20	**0.24**	0.003
TCDH024	**Haib West**	**778074**	**6823190**	**653**	**358.44**	**10**	**-60**	**No significant Intersections**				
TCDH025	**Deposit**	**781435**	**6822603**	**528**	**568.7**	**360**	**-90**	0	568.7	568.7	**0.22**	0.005
Incl.								71.7	122.4	50.7	**0.21**	**0.013**
Incl.								402	480.6	78.6	**0.32**	0.005
Incl.								20	50	30	**0.36**	0.003
TCDH026	**Deposit**	**781024**	**6823563**	**495**	**475.33**	**192**	**-70**	0	475.3	475.3	**0.3**	0.007
Incl.								349	427	78	**0.31**	**0.021**
Incl.								178	327	149	**0.57**	0.004
Incl.								283	313	30	**0.81**	0.007
TCDH027	**Deposit**	**781528**	**6823066**	**409**	**446.07**	**10**	**-60**	0	446.07	446.07	**0.2**	0.006
Incl.								207	262	55	**0.27**	0.004
Incl.								0	87.5	87.5	**0.31**	**0.013**
TCDH028	**Deposit**	**780804**	**6823001**	**442**	**401.34**	**360**	**-90**	0	401.14	401.14	0.15	**0.010**
Incl.								20	182	162	**0.27**	**0.014**
Incl.								146	176	30	**0.45**	**0.023**
TCDH029	**Deposit**	**780816**	**6822996**	**458**	**200.11**	**185**	**-60**	0	200.11	200.11	**0.27**	0.005
Incl.								71.1	125.5	54.4	**0.4**	0.004
Incl.								90	99.2	9.2	**0.75**	0.007

(See continuation of Table 10-2 on next page)

Table 10-2 continued

Hole ID	Target	X	Y	Z m	Length m	Azimuth	Dip	From (m)	To (m)	Interval (m)	Cu %	Mo %
TCDH030	**Deposit**	**781309**	**6823077**	**457**	**200.51**	**12**	**-65**	0	200.51	200.51	**0.29**	**0.013**
Incl.								39.7	92.7	53	**0.41**	**0.012**
Incl.								103.22	200.51	97.29	**0.30**	**0.018**
Incl.								51.3	77	25.7	**0.66**	**0.019**
TCDH031	**Deposit**	**781432**	**6822985**	**435**	**200.47**	**15**	**-60**	0	200.47	200.51	**0.27**	**0.010**
Incl.								62.9	164.2	101.3	**0.36**	0.008
Incl.								81.27	121.8	15.67	**0.49**	**0.260**
Incl.								51.3	137.47	9.57	**0.54**	0.007
TCDH032	**Deposit**	**781900**	**6822630**	**408**	**200.1**	**188**	**-60**	0	200.1	200.1	**0.26**	0.002
Incl.								4	37.3	33.3	**0.33**	0.002
Incl.								77	82	5	**0.45**	0.004
Incl.								110	117	7	**0.58**	0.004

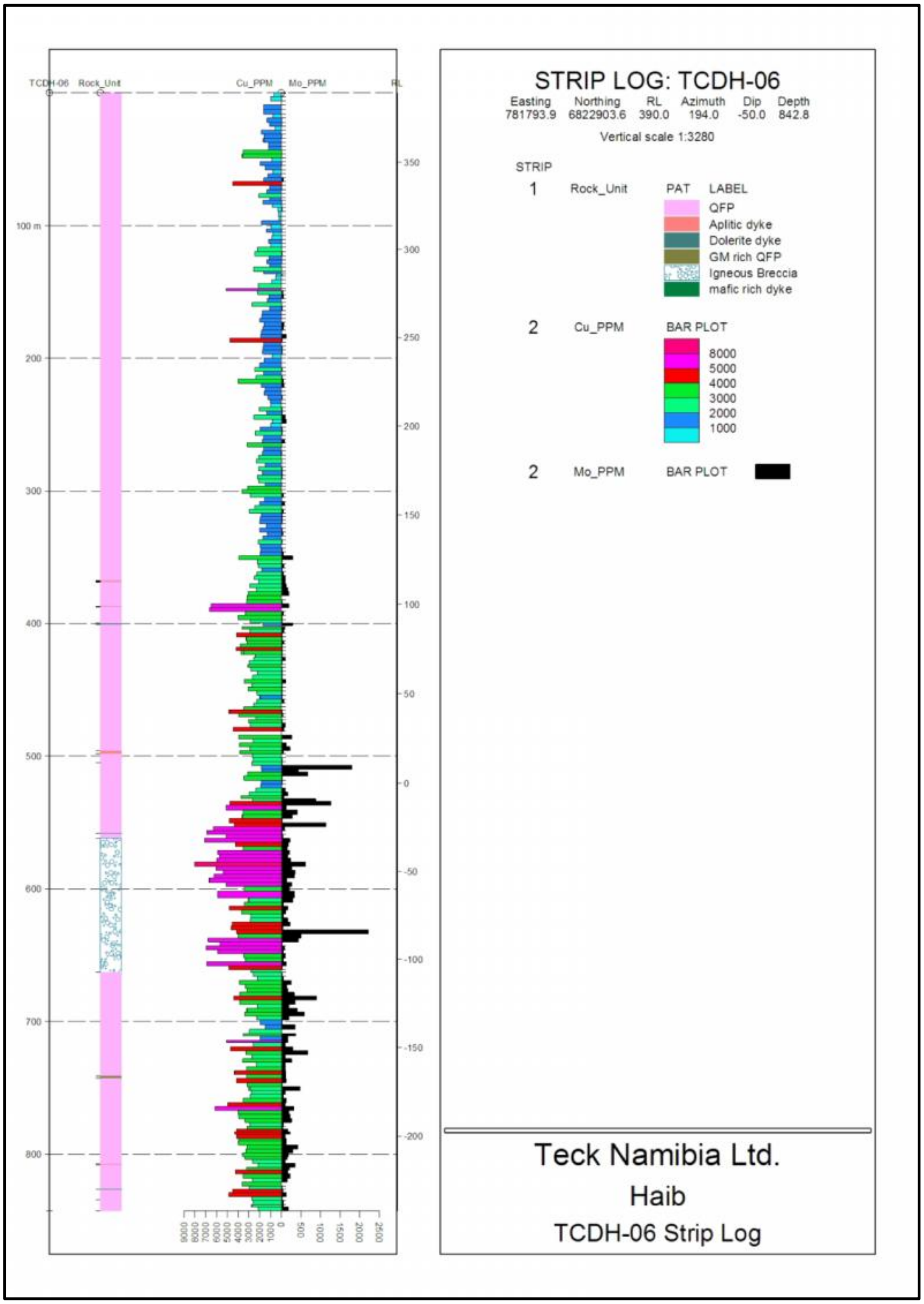


Figure 10-3: This diagram shows the strip log for drillhole TCD-06, the deepest hole drilled at Haib by Teck. This drillhole was located just west of the centre of the main deposit at one of the lowest points (in the Volstruis riverbed). It can be seen that sections of this drillhole below 500m returned significantly high copper values and above average Molybdenum values (Source: Teck, 2012[42]).

11. SAMPLING PREPARATION, ANALYSIS AND SECURITY

11.1 Historical Sampling

The historical drilling database comprises physical details of each hole, a drill lithological log, details of sampling intervals and assay results from approximately 25,000 samples of which the vast majority are 2 metre half-core composite samples from the Rio Tinto drilling (22,800 samples). The King Resources composite samples averaged 4.5 metres average length, while the Falconbridge samples were an average 3.0 metre length.

Of the total samples approximately 15,000 have values greater than 0.1% Cu but only 1,100 have values greater than 0.5% Cu. The acid soluble oxide-copper database comprises 1,980 samples.

Specific gravity ("SG") measurements were carried out by RTZ on 40 drillholes giving approximately 7,000 determinations; SG's ranged from 2.43 to 3.35 and averaged 2.71; GFM continued the process of SG determinations on core samples during their drilling campaign, sampling every tenth sample.

It is not possible for me (Peter Walker) to comment on the sample preparation, analysis and security of these historical drill samples as the details of quality control and assurance and copies of original assay certificates are not available. It is known that the RTZ samples (22,800) were all prepared on site, Rio Tinto having a prep-laboratory at the campsite fitted with crusher, pulveriser and splitters – the dust extractor plus parts of the other equipment are still on site. It is believed that the actual analyses were done off site at both the RTZ Rossing mine and RTZ Palaborwa mine laboratories.

11.2 Teck Sampling

No core sampling was being carried out at the time of the site visits so I (Peter Walker) was unable to verify or review the Teck sampling procedures. I (Peter Walker) have been supplied with an internal Teck memorandum [39] detailing the sample preparation protocols to be employed during both core and geochemical sampling at the Haib project and I (Peter Walker) have been assured that these protocols are strictly enforced on site and at the independent prep-laboratory in Windhoek (Analytical Laboratory Services) and the independent assay laboratory (Acme Analytical Laboratories – www.acmelab.com, now a subsidiary of Bureau Veritas) ("Acme") in Vancouver, Canada. The protocol lists the following important steps: -

- A standard sample to monitor analytical accuracy, a field blank sample to monitor carry-over contamination at the crusher and a core (or soil/stream/rock) duplicate sample to monitor geological, preparation and analysis variation are to be inserted in the core-shed every 20 samples. The appropriate standards used at the Haib are sourced from CDN Resource Laboratories in Canada who have supplied certificates certifying the material supplied. I (Peter Walker) have had sight of a selection of these certificates and am satisfied as to their veracity and appropriateness in terms of the range of expected values for copper, gold and molybdenum.

- All drillholes are sampled from the start to end of hole; the core is split using a water lubricated diamond saw blade which is cleaned frequently by using a brick to prevent carry-over contamination. Core samples are bagged in good quality plastic bags to avoid contamination or loss of fine material during transport. Sequential sample numbers are assigned and recorded on the paper drill log sheet. All of the hole's survey, logging and sampling data are captured and stored in a secure database system (Excel) on a laptop in the field and backed up by transfer to a central Access database system in Windhoek. All data is subject to routine validation during capture and storage. Drill log sheets, survey records and drill assay certificates are all securely filed in Windhoek on a regular basis.

- At the independent sample preparation laboratory in Windhoek the entire sample is dried, crushed and check screened to ensure that at least 80% of the crushed material passes through a 2mm screen; the entire crushed sample is riffle split to approximately 1Kg and this is pulverized in a disk mill as a single charge with testing of the pulp to ensure that a minimum 80% is <75 microns. Every 20 samples a duplicate sample is drawn off of the riffler to assess combined preparation and analytical variation. All of the sizing tests are recorded in a book to ensure compliance. Samples that do not pass the sizing tests are re-crushed or re-milled until a pass is obtained. The preparation laboratory cones and quarters the pulp sample to obtain a 100gm of material which is bagged in a good quality paper envelope. The entire remaining crushed and pulped sample is retained and stored by Teck so that umpire samples may be taken.

- The drill core assays routinely include copper, molybdenum, gold and 21 additional elements all determined by an ICP-ES technique.

- The Acme Vancouver facility has maintained a quality system compliant with the International Standards Organization (ISO) 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories. In October 2011 the Vancouver facility received formal approval of its ISO/IEC 17025:2005 accreditation from Standards Council of Canada.

- The Analytical Laboratory Services facility in Windhoek is not certified as being ISO 17025 compliant. QA is provided by replicate analysis, the insertion of control samples, the submission of samples to independent laboratories in Namibia and the participation in independent proficiency testing schemes.

Teck have stated on public record that: -

"The design of Teck's drilling programme, quality assurance / quality control programme and the interpretation of results are under the control of Teck's geological staff. The QA/QC programme is consistent with industry best practices. Drill core is logged and cut onsite, with half-core samples prepared at Analytical Laboratory Services, Windhoek, Namibia. Prepared samples are shipped to Acme Analytical Laboratories, Vancouver, Canada for appropriate base metal assaying and gold fire assaying techniques. All analytical batches contain appropriate blind standards, duplicates and blanks inserted at regular intervals to independently assess analytical accuracy and precision."

11.3 Sample Security

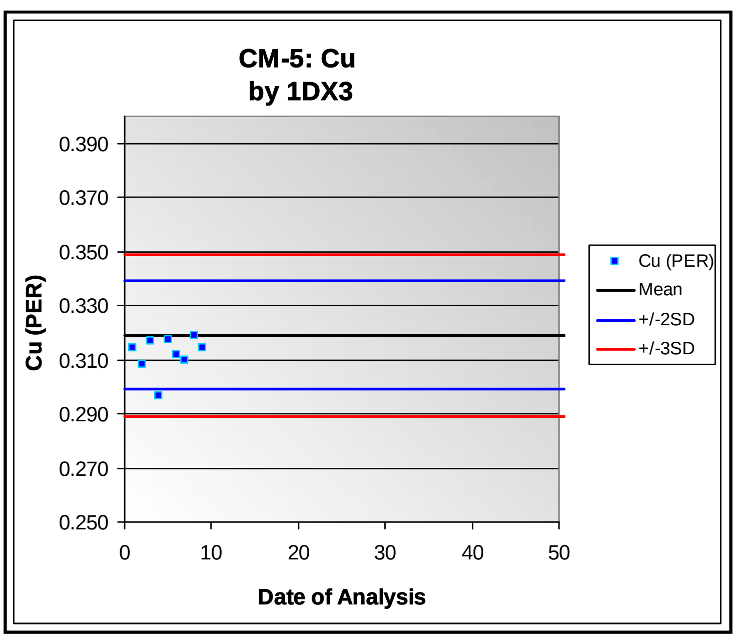
The core yard at the Haib camp is surrounded by 2m wire fencing and the metal entrance gate is secured by a padlock. I (Peter Walker) am assured by the site geologist that sampling of core is done under his supervision; the bags are secured immediately after the cutting and sampling process and the samples taken are stored within the locked RTZ laboratory building within the core yard until transported by him to Windhoek.

11.4 Data Verification

The Teck guidelines for data verification are as follows:

- The guideline for Standard failure is: Any Standard sample which falls outside of the mean +/- 3 standard deviation range or any two consecutive sample results outside of the mean +/- two standard deviations range.
- Re-analysis is at the discretion of the geologist, but the guideline is that any failure (as with CM-5 below) should trigger re-assay of all samples from the first sample after the previous passing Standard sample to the sample previous to the next passing Standard.

An example of Teck's graphical plots, this one for Standard Sample No.CM-5 is given below:



11.5 Contributing QP's Comments on Sample Preparation, Analysis and Security

In our (Peter Walker and Dean Richards) opinion Teck's knowledge regarding the controls on high grade zones ensures that there is no bias in their sampling. On the assumption that Teck personnel strictly adhere to their protocols regarding sample collection, transport, preparation, security and analytical procedures, then the reliability, validity and integrity of the sample assay results should be assured. Assuming that Teck personnel are adhering to their stated procedures, the chain of custody in sample collection and transport would be well controlled.

Teck used duplicates, standards and blanks to check the accuracy and precision of their assay data. The amount of QC / QA data is significant and the spreadsheet files and graphical presentation of their results have been check sampled by us (Peter Walker and Dean Richards) and found to be adequate to ensure veracity of their results.

In the contributing authors' (Peter Walker and Dean Richards) opinion there is no relationship between Analytical Laboratory Services in Windhoek and/or Acme and/or Teck as operator for HM apart from a normal principal and client business relationship and both laboratories can be classified as independent applying all of the standard tests of independence.

12. DATA VERIFICATION

12.1 Historical Data

Original assay laboratory sheets or certificates were not located for the Falconbridge, KRC, or RTZ data. In addition, there were no records of any assay duplicates, field re-splits or check assays having been carried out by independent laboratories.

The RTZ drill samples were collected as composite half core samples over 2m sampling intervals and a total of some 45,865 metres has been assayed. Validating this database has been difficult because the assays were done at the RTZ Rossing laboratory with every tenth sample check assayed at the RTZ Palaborwa Mine laboratory. No original or copies of assay certificates have been located to validate the historical database. In order for HM to utilize the RTZ data in any future resource estimate, re-assaying of important intervals of RTZ core is required; this programme has been implemented – see discussion under section 10.1 above.

The NCJV drilling (completed after the Behre Dolbear historical estimates), supported the mineralisation models created from previous assay results but could not verify them.

RTZ also prepared extensive metallurgical composites comprising sequential down-hole samples over approximately 20 metres. A historical estimate carried out by GFM based solely on this composite data gave comparable results to estimates using the other drill assay data.

12.2 Teck Resampling

Due to the difficulty of validating the previous drilling, and in particular the RTZ database, in 2010 and 2011 Teck re-logged and re-assayed 619 x 6m quarter-core composites of the RTZ drillhole cores from 14 drillholes (see Figure 10-1 above) representing approximately 8% of the RTZ assay data and could potentially extend this programme of RTZ core analysis so that it can be included in a future compliant resource estimate – please see our (Peter Walker and Dean Richards) discussion of the comparative results in Section 10.1 above. RTZ only assayed for copper on a systematic basis while all of the Teck assays routinely include copper, molybdenum, gold and 21 additional elements all determined by an ICP-ES technique.

12.3 Assessment of Quality Control Data

Obsidian Consulting Services conducted a review of the QA/QC programme implemented by Teck using the certificates of analysis received from Acme Labs and provided by Mr Neil Grumbley of Teck. This review compared the results of field duplicates, blanks as well as the various standards utilised with respect to Cu and Mo.

12.3.1 Duplicates

Two types of field duplicates were utilised viz. core samples and crusher duplicates. Figure 12-1 shows scattergrams of the original core sample Cu and Mo grades versus the duplicate grades. The left-hand charts show the full data extents while the right-hand charts zoom into the lower grade areas. From Figure 12-1, it can be seen that most of the core duplicates for Cu fall within the ±10% limits from a correlation of one and that 14 of the 415 (3%) core duplicates fall outside of the ±25% limits. The R^2 (Pearson's Coefficient) of 0.94 confirms the sampling procedures for the core sampling and the reproducibility of the associated results to be of a good quality.

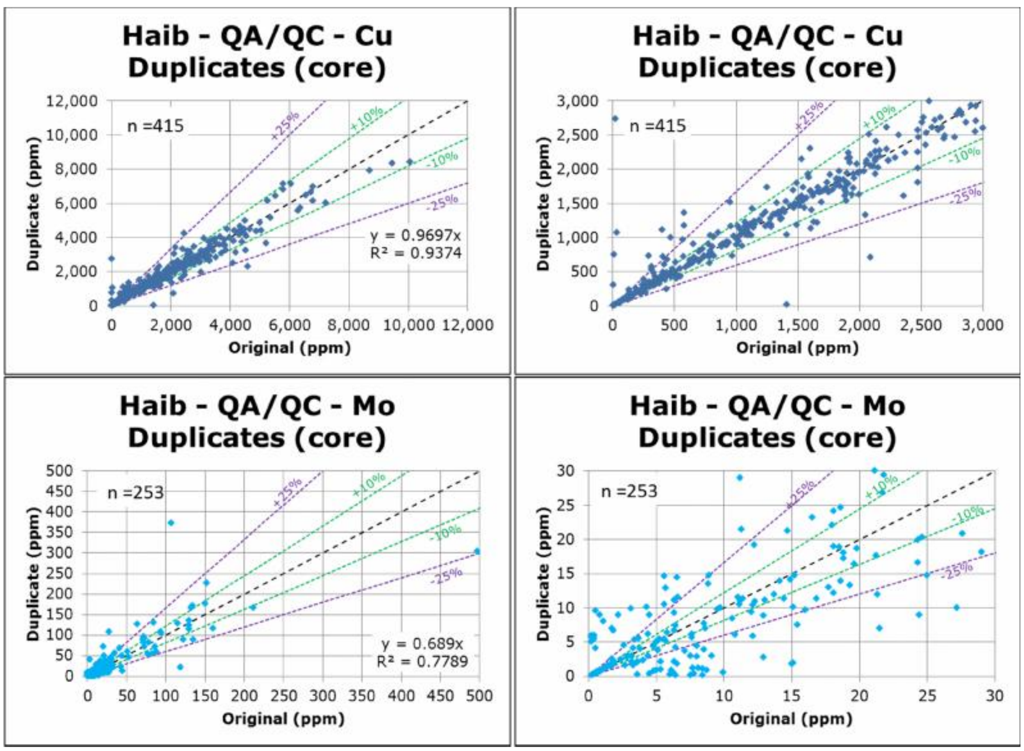


Figure 12-1: Quantile-Quantile plots comparing core original assays and core duplicates. The right-hand graphs have zoomed in near the low-grade samples as indicated by the X axes values.

This is less true for the Mo results. While generally the bulk of results fall within the ±25% limits, a relatively low R^2 of 0.7789 is borne out by the fact that 63 of the 253 (25%) duplicates for Mo lie outside the ±25% limits. Most of these lie at a Mo grade of <10ppm which means the assays here have been undertaken right at the limits of assay ability and these results are to be expected to some degree. Any impact is not material at the resource scale though and it is likely this can be attributed to a "Fundamental Error" being a function of the contained grade.

The duplicates produced at the crushing stage were then analysed and are presented in Figure 12-2 below. Again, the Cu results point to good work within the crushing procedures as shown by the R^2 of 0.98 and only 5 samples (1%) falling outside the ±25% limits ±25% limits. The Mo results again are not as good as the Cu results, however most of the failures for Mo are at concentrations below 10ppm which is right on the edge of practical constraints during the crushing procedure.

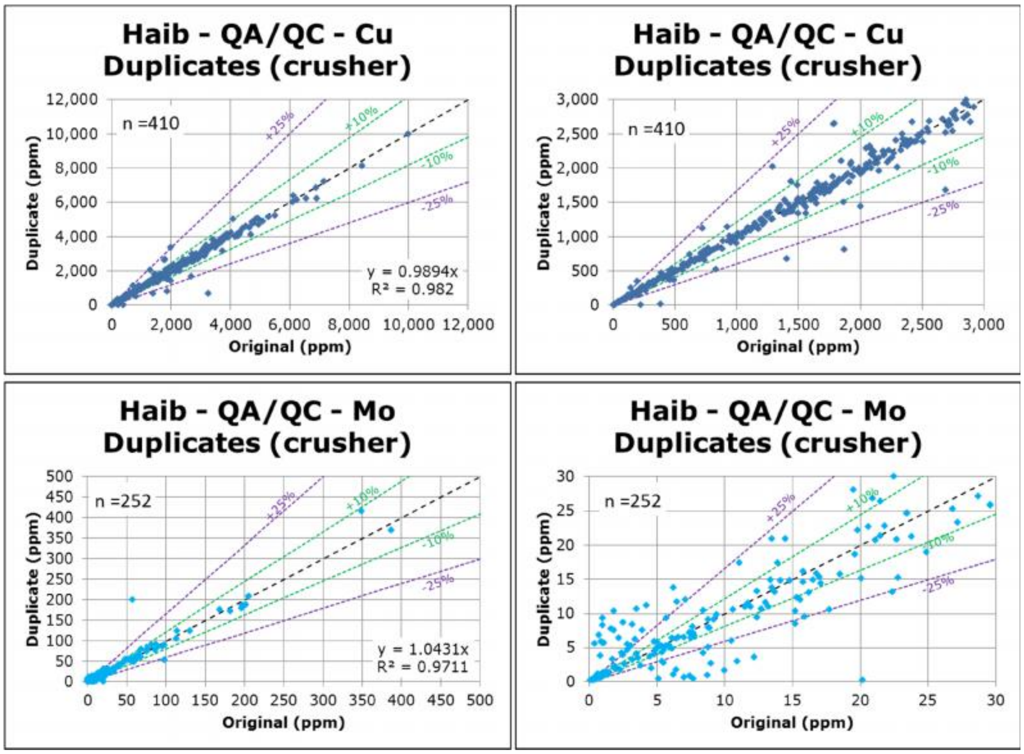


Figure 12-2: Quantile-Quantile plots comparing crusher original assays and crusher duplicates. The right-hand graphs have zoomed in near the low-grade samples as indicated by the X axes values.

12.3.2 Blanks

Figure 12-3 below shows a line graph (black) of the Cu and Mo grades received for submitted blanks along with the average grade of the batch (red shading) with which the blank was analysed. The purpose of these is to highlight erroneous results and quantify them relative to the batch results.

In the case of the Cu results, there were a couple of significant failures indicated in Figure 12.3 by labels of the respective sample numbers; there are only 13 failures from 415 blanks (3%) which is indicative of little or no contamination during sampling or during analysis. One issue though is highlighted by the blue shading where a number of sequential blanks failed. It is not known whether this is due to contamination during sampling in the field, instrument calibration or even an issue of homogeneity with the blank material but Teck’s procedures would have highlighted the matter and it is assumed the issue was examined in detail at the time and resolved.

With respect to Mo, there are a couple of noticeable issues (see Figure 12-3 below). Initially, the blanks returned values of between zero and about 5ppm (①). For some reason at about halfway through the programme, this range changes to between zero and 30 with a mean at about 10ppm (②). In some cases, the values returned by the blank are higher than the batch mean which raises some concerns about the quality of the Mo grades received.

Possible causes of this include;

- Instrument calibration
- Blank material not suited for Mo
- Crushing chamber not being adequately cleaned between samples. This is certainly feasible for Mo as molybdenite is malleable and ductile and can be “smeared” on internal surfaces during crushing.

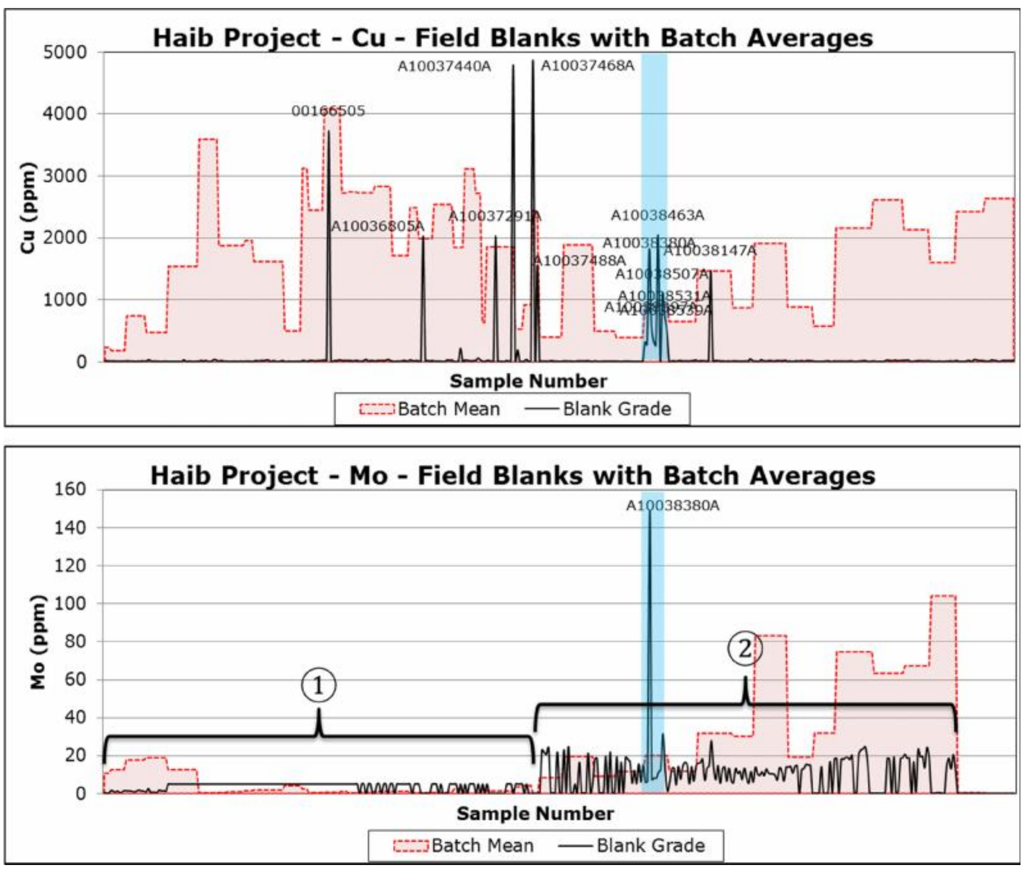


Figure 12-3: Line graphs of blank results for Cu and Mo comparing them to Batch Mean’s and ordered by sample number

12.3.3 Standards

Thirteen different mineral standards were used by Teck as part of their QA/QC programme. The summary statistics of the standards are tabulated in Table 12-1 below. Note that only certified values are shown.

Regarding Cu assays, the CGS and CM standards cover a range of values from 0.112% to 0.725% which is appropriate for the grade ranges encountered at Haib. The Relincho standards comprise two very low-grade standards (ST1 and ST2) and one further standard with Cu grade approaching 1%.

Regarding the Mo assays, the standards cover a low of 0.004% (40ppm) to 0.05% (500ppm). Again, the available range is generally suited to the Mo grades found at Haib. A summary of the results of the standards is provided in Table 12-1. Generally, pass rates exceed 90% with only the Relincho ST-3 standard raising some issues with only 2 of the 4 samples submitted passing. The Mo standard results are generally good.

Table 12-1: Summary of the certified standard materials used by Teck.

		Certifed Values Only						Results					
		Cu(%)		Au (g/t)		Mo(%)		Cu			Mo		
		Mean	2SD	Mean	2SD	Mean	2SD	No. Samples	Failures	%Pass	No. Samples	Failures	%Pass
CGS-16	4-acid	0.112	0.005	0.140	0.046			69	5	93%			
CGS-22	4-acid	0.725	0.028	0.64	0.06			14	1	93%			
CGS-23	4-acid	0.182	0.010					75	4	95%			
CGS-24	4-acid	0.486	0.034	0.487	0.05			43	3	93%			
CM-4	4-acid	0.508	0.025	1.18	0.12	0.032	0.004	28	2	93%	16	2	88%
CM-5	4-acid	0.319	0.020	0.294	0.046	0.050	0.005	41	6	85%	11	0	100%
CM-7	4-acid	0.445	0.027	0.427	0.042	0.027	0.002	16	0	100%			
CM-16	4-acid	0.184	0.014			0.016	0.002	45	1	98%	34	2	94%
	Aqua regia	0.184	0.016			0.016	0.003						
CM-20	4-acid	0.316	0.016			0.030	0.002	32	0	100%	28	2	93%
	Aqua regia	0.314	0.014										
CM-21	Fire Assay			0.467	0.052								
	4-acid	0.527	0.022			0.036	0.002	9	0	100%	7	2	71%
	Aqua regia	0.530	0.028										
Relincho ST-1	Aqua regia	0.018	0.002										
	3-acid	0.019	0.002										
	4-acid	0.018	0.002					13	0	100%			
Relincho ST-2	Aqua regia	0.076	0.006										
	3-acid	0.076	0.008			0.004	0.000						
	4-acid	0.075	0.008					21	0	100%			
Relincho ST-3	Aqua regia	0.835	0.046			0.016	0.002						
	3-acid	0.823	0.034			0.016	0.002						
	4-acid	0.832	0.032			0.016	0.002	4	2	50%			

Figure 12-4 below provides plots of the results of standards CGS-16, CGS-23 and CGS-24. From this Figure, it can be seen that while the reported values lie within the ±3 Standard Deviation range applied by Teck, most of the values lie below the certified mean. For CGS-16, 80% are below the mean, while 83% of CGS-23 samples and 63% of CGS-24 samples respectively are below the certified mean.

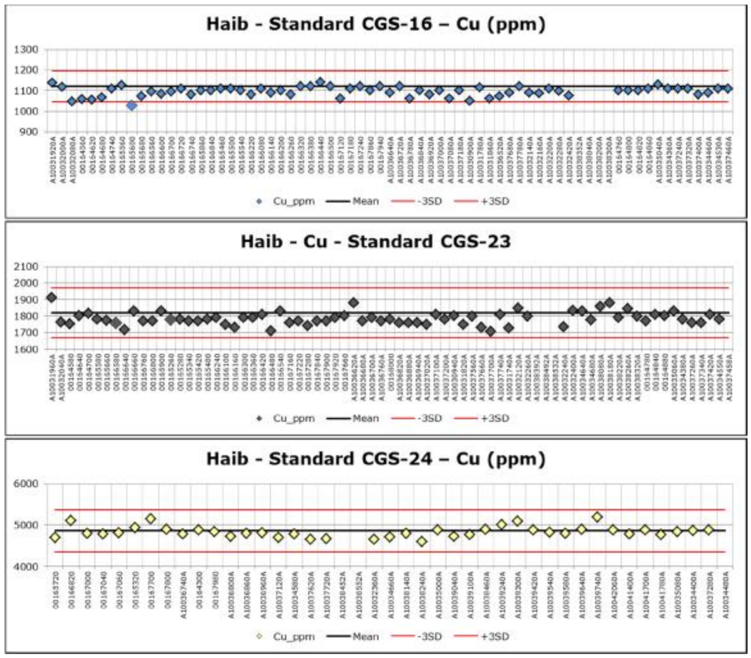


Figure 12-4: Plots of standard results for Standards CGS-16; CGS-23 and CGS-24 ordered by sample number.

This implies a potential instrument calibration error (though minor scale). Depending on the standard used, this error varies between 0.5% and 5%. As these standards account for ~46% of the total standards submitted, it can be inferred that nearly half the Cu grades are underestimated by somewhere between 0.5% and 5% i.e. percentages of their reported values and not whole numbers.

12.4 Contributing QP's Overall Interpretation of the QA/QC Programme

The QA/QC program implemented by Teck was aligned with international standards and has generally delivered good results. The field duplicates show that sampling practises employed by Teck have produced accurate, repeatable results especially for Cu. Due to the low Mo grades involved, greater deviation is present though this is expected due to the fact that the sampling is being done so close to the lower detection limits for Mo.

The field blanks show that as a rule, contamination of Cu samples has been kept low by the sampling practices. The selected blank was appropriate for Cu and exceptions are readily visible. There do appear to be some issues with respect to blanks for Mo. Mo values returned for the blank material are commonly a significant proportion of the average Mo grade of the associated batch. It is not known whether this is due to an inappropriate blank for Mo or laboratory issues but it does cast some suspicion on the Mo grades.

A further benefit of the QA/QC programme utilised by Teck is that it provides a basis for comparison to the historical data to determine the quality of this data. This was done as follows. Boreholes with QA/QC data were tagged separately from those without QA/QC and were superimposed on the resource model. This is shown in Figure 12-5 where boreholes with QA/QC are shaded red and those without, black. From Figure 12-5, it can be seen that the two sets are relatively well interspersed particularly in the central portion of the deposit.

Within the shaded area in Figure 12-5, the two sets of drilling have sampled approximately the same sample space. The distributions of Cu samples (in ppm to provide more resolution for comparison) with QA/QC were compared to those samples without QA/QC. This was done using a QQ Plots which is shown in Figure 12-6. The reasoning behind this was that if there is no bias between the two sets of data, the curve should plot on or near the 45° dashed line depicting perfect correlation.

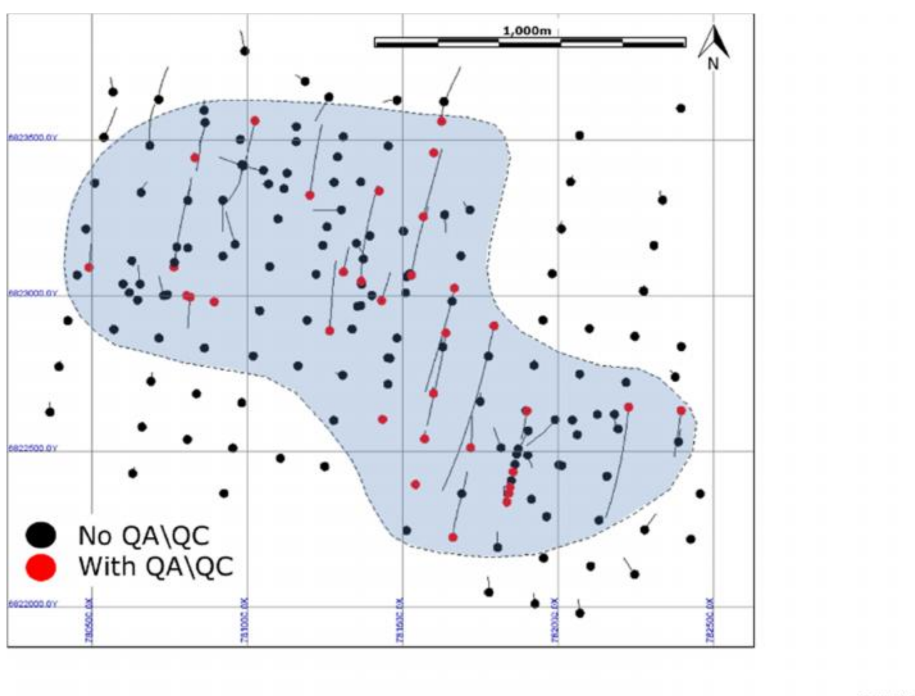


Figure 12-5: Plan showing the Haib resource 3D models and the associated drilling data coloured according to whether completed under QA/QC or not.

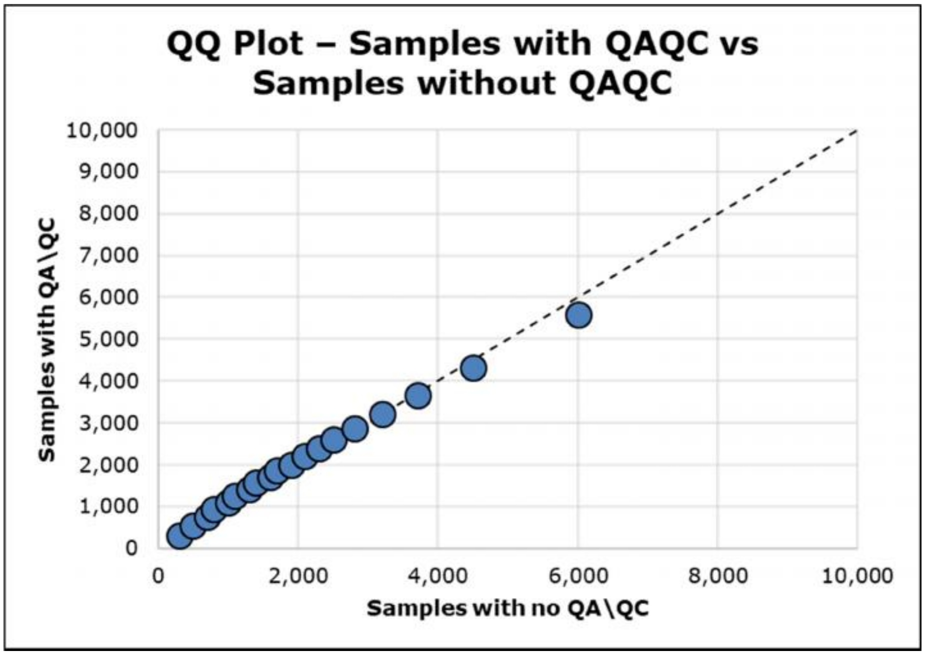


Figure 12-6: QQ Plots comparing by resource domain, the distributions of Cu results from samples taken under an appropriate QA/QC regime and those that weren't.

A look at Figure 12-6 shows that there is excellent correlation between the two. For this reason, it is the opinion of Obsidian Consulting that the historical data, despite their lack of QA/QC can be merged with the Teck dataset for use in mineral resource estimation without concerns about the introduction of any material bias.

13. MINERAL PROCESSING & METALLURGICAL TESTING

13.1 Metallurgical Testwork

13.1.1 Historical Metallurgical Testwork – 1990's

In early 1996 NCJV commissioned engineering firm Davy & Minproc (in joint venture) to prepare a feasibility study for the Haib Copper Project [17].

The first phase of the feasibility study involved an extensive programme of metallurgical test work; to accommodate this test work, a sampling programme involving diamond drilling and the excavation of an adit (photograph 2) and cross-cuts into a representative section of the defined mineralisation was completed. This involved some 150 metres of underground development of an adit at a nominal 2 x 2 metre cross-section with two short cross-cuts at the end of the adit. This adit and cross-cut generated some 2,000 tonnes of fresh material for metallurgical test-work. The adit intersected higher grade material delineated by RTZ's close-spaced drilling on section 000E/W. The 2,000 tonnes of rock were removed from the adit, stacked in heaps representing each 1m advance of the excavation and then sampled. This resulted in the accumulation of a representative bulk sample of some 500 tonnes which was sent to various laboratories for test-work. The balance of 1,500 tonnes is still stockpiled on site (see photograph 8).

Photograph 8:
The remaining metallurgical bulk sample stacked in separate heaps.

Test work was done at the following laboratories: -

- Mintek, Johannesburg, South Africa.
- University of the Witwatersrand, Johannesburg, South Africa.
- Metcon, Tucson, Arizona, USA.
- Amdel, Adelaide, S. Australia.
- Ammtec, Perth, W. Australia.

Test work included: -

- Mineralogy.

- Flotation.
- Comminution.
- Roasting & Acid Leaching of concentrates.
- Autogenous and semi-autogenous milling.
- Bacterial oxidation of concentrates.
- Column leach test work for heap leaching.

The reports generated by this study are listed in the References and Bibliography section of this Report numbered 10 to 17 and the results are summarized in the Davy-Minproc phase II feasibility study [17] and the NCJV Feasibility Study - Executive Summary report [9]. The results of these various studies are summarized in the following paragraphs.

The basic initial test work results showed that the Haib mineralisation is a competent quartz feldspar porphyry rock having a ball mill work index of between 17 and 20. The copper mineralisation is primarily chalcopyrite which is highly amenable to flotation. The test work indicates that grinding to 80% passing 150 microns will yield an overall Roast Leach Electrowin ("RLE") recovery to cathode copper of 83.7%.

The second phase test work involved the design and costing of a 34.2Mtpa RLE plant with an associated 14Mtpa heap leach operation reported on in the Davy-Minproc report [17]. This showed a RLE treatment cost at that date of US$2.36/tonne plus a contingency of 10%. At then current comparable rates, this was regarded as a low-cost metallurgical operation; because of this low cost and successful conventional RLE treatment process the NCJV management decided not to incorporate heap leaching in further studies despite the good results achieved in column test work at the Metcon Laboratories in Arizona.

The NCJV then embarked on definitive engineering studies to design and cost all the mining and metallurgical plant and equipment required to mine at Haib, as well as preliminary environmental studies [24 to 33] prior to definitive drilling of the selected open pit area and then the production of a final bankable feasibility study. The NCJV then ran into financial difficulties and the further test work was abandoned.

13.1.2 Modern Metallurgical Testwork

In 2003, at the request of Mintek, the then claim holder Mr. George Swanson provided a 1-tonne sample of oxide mineralisation and a 1-tonne sample of sulphide mineralisation from Haib so that Mintek could do further testing of their new proprietary heap bio-leach process [14].

The oxide sample was crushed to -25mm and blended to homogenize it before sub-sampling. Sub-samples were submitted for chemical assay, mineralogical study and sieve size analysis. Roll bottle tests on samples determined the acid consumption characteristics of the mineralisation and the particle size of copper leach kinetics. Column test work to determine agglomeration requirements and percolation tests at various irrigation rates were completed.

The results indicated that: -

- The oxide copper sample contained 3% Cu.
- The copper is present as acid-soluble silicates and carbonates.
- Copper extraction of 70% to 93% is possible with acid consumption of 1.9 to 2.4kg acid/kg Cu.
- The feed acid concentration should be kept at 10g/l for maximum extraction.
- Highest extractions were obtained at -12mm & -6mm crush sizes.

The sulphide sample was treated in a similar fashion and the results indicated: –

- The mineralised sulphide sample contained some 0.6% copper.
- The copper is present as chalcopyrite associated with pyrite.
- The mineralised sulphide is difficult to agglomerate & pelletising was tried.
- Higher temperatures and lower crush sizes improved the leach kinetics.
- The best extraction for 6mm material at 65^{0}C was 80% Cu recovery over a 200-day period.

Neither Mintek nor Swanson has made any economic assessments of these test results.

13.1.3 Metallurgical Testwork Review

HM commissioned METS to examine the latest metallurgical process technology and review the historical test work and then develop conceptual ideas for processing options. METS have been commissioned to complete a Preliminary Economic Assessment using the results of their processing study and the Resource Estimates completed by Obsidian Consulting Services.

METS Engineering has completed the mineral processing and metallurgical study (52) that considers amongst others:

- Comminution
- Heavy Liquid Separation
- Bio-Heap Amenability
- Flotation

Four recovery options were considered for economic evaluation: -

- Option 1: Ore sorter upgrading, dense media upgrading, flotation and heap leaching of the tails.
- Option 2: Two-stage dense media upgrading, flotation and heap leaching of the tails.
- Option 3: Ore sorter upgrading and heap leaching of the upgraded material.
- Option 4: Whole ore heap leaching.

13.2 Contributing QP's Comments

The NCJV adit was located specifically to test a cross section of the oxide mineralisation as well as normal grade and high grade sections of the defined copper / molybdenum sulphide mineralisation; as such, the metallurgical samples can be regarded as being closely representative of all of the mineralisation at Haib and none of the metallurgical or feasibility reports indicate anything to the contrary; neither RTZ, NCJV or Teck have shown the presence of any deleterious elements that could have a significant effect on potential economic extraction.

Teck considered further metallurgical test work and drilled 4 diamond drillholes totalling 801.19m of HQ diameter (63.5mm) core with the intention of taking samples from the high-grade zones to be able to verify the metallurgical characteristics and perhaps for testing the amenability of the material to various (and more modern?) copper extraction techniques. Teck, however, did no further metallurgical testwork.

HM have commissioned this PEA which has made use of the historical metallurgical test work and the positive result of the PEA will probably result in HM commissioning further metallurgical studies on both existing adit samples, new adit samples, Teck's drillhole samples and any new metallurgical drillhole samples from a yet-to-be planned mining area.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

In July 2017, Obsidian Consulting Services, an independent geological consultancy, conducted a mineral resource estimate for the Haib Copper Project using the outputs of some 3D modelling work that had been completed by Teck using the LeapFrog GEO software package. The models were analysed with respect to their grade distributions and an appropriate domain was selected on which the mineral resource estimate was then based. A mineral resource classification based principally on data density was applied to derive a mineral resource statement.

14.2 Source Data

14.2.1 Drillhole Data

All the available drill hole data for Haib was compiled in a single Geovia-GEMS project. The summary statistics of the complete, compiled drill hole database are given in Table 14-1. Of significance from this is the fact that Cu assays outnumber Mo assays by more than 3:1 while the deepest intersections achieve a depth of more than 800m below the surface topography.

Table 14-1: Summary drilling statistics by drilling programme

Series	No. Holes	Suitable for Estimation	Total (m)	Average m/hole	Max. Depth	Cu Assays	Mo Assays	
							Assayed	Visual
ADIT01	1	1	126.00	126.00	40	63	63	
GFMHB01 - GFMHB12	15	15	4,726.40	315.09	464	2,186	2,034	
H01 - H12	11	11	1,010.72	91.88	225	253	0	
HB001 - HB210A	121	121	45,795.15	378.47	653	22,838	1,530	1630
KO1 - KO4	3	3	151.49	50.50	49	34	0	
KS01 - KS21	18	18	3,324.76	184.71	288	727	0	
TCDH-01 - TCDH032	32	32	14,252.93	445.40	796	5,999	5,999	
	201	201	69,387.45	345.21		32,100	9,626	1,630

The positions of the drill holes relative to the modelled portion of Haib are given in Figure 14-1. The borehole collars are coloured as to whether they were subjected to a QA/QC programme (red) or not (black).

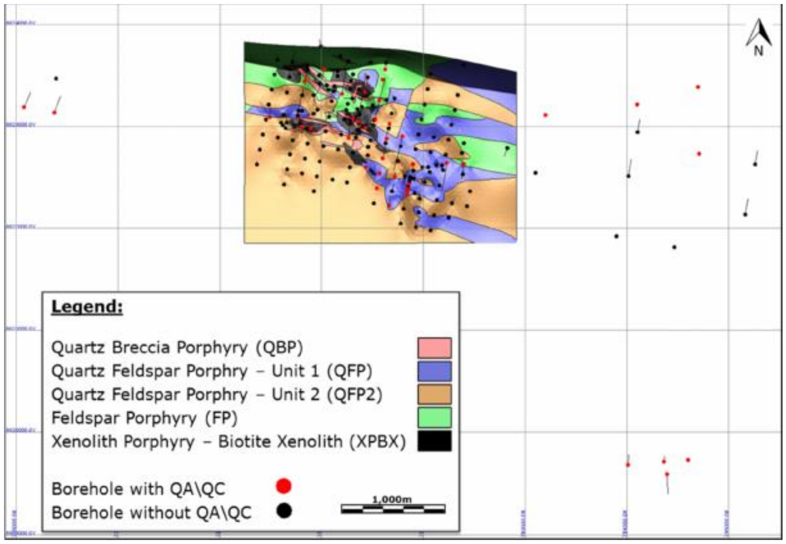


Figure 14-1: Plan showing the limits of the geological modelling conducted by Teck with the available drilling overlaid.

14.2.2 Three Dimensional Models

A summary listing of the received 3D models from Teck is given in Table 14-2 while Figure 14-2 shows an isometric view of these. The geological model comprises major faults as well as lithological models. The Cu grade isoshells were provided, the first approximating a 0.3% grade limit, the second 0.2%. An isoshell of Mo grades elevated above background levels was also received.

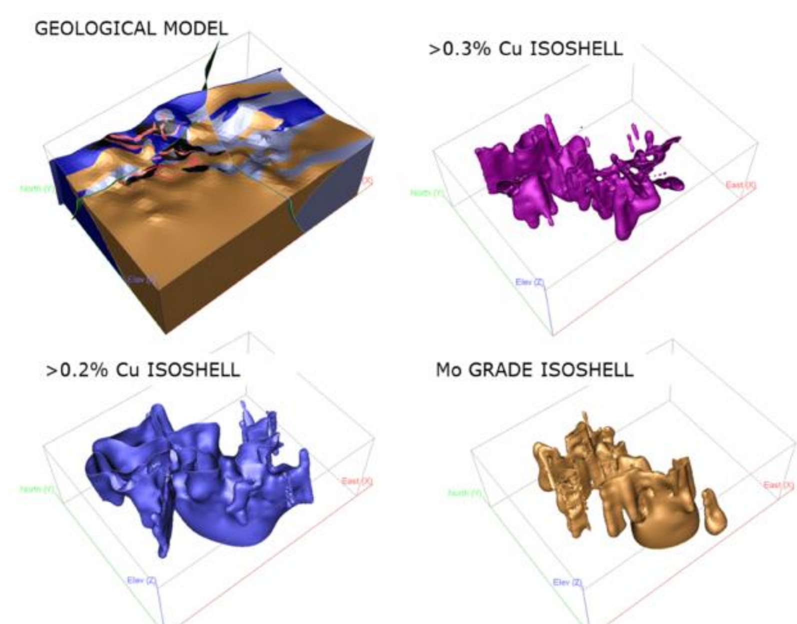


Figure 14-2: Isometric view showing the various 3D models received from Teck.

Table 14-2: Listing of files received from Teck

Type	LeapFrog File
Topography	Haib_Topography.dxf
Structural Model	GM_Lithology -EW FAULT.dxf GM_Lithology -NorthShear.dxf GM_Lithology -NS QV.dxf
Geology	GM_Lithology -FP.dxf GM_Lithology - QBP.dxf GM_Lithology - QFP.dxf GM_Lithology - QFP2.dxf GM_Lithology - XPBX.dxf
Grade Isoshells	GM_GradeOutlines - High Grade.dxf GM_GradeOutlines - Low Grade.dxf GM_GradeOutlines - Mo Outline.dxf

14.3 Domain Selection

Each of the solid models received represents a potential domain for resource estimation and reporting. The univariate statistics were calculated for each and are shown in Table 14-3. The Mo mineralisation isoshell was not considered as Mo was viewed as secondary relative to Cu. Of the lithological models, the QBP and XPBX show the highest mean and median grades, followed by the QFP's and then the FP. Notwithstanding this, the differences in grade are muted with each showing similar ranges in Cu grade. This implies that rather than being restricted to a single domain, the Cu grade is distributed throughout the deposit.

Table 14-3: Summary univariate statistics by domain

	Cu (%)							
	All	FP	QBP	QFP	QFP2	XPBX	>0.3 Cu	>0.2 Cu
Count	32,100	2,929	2,074	6,808	10,159	3,679	5,738	13,969
Minimum value	0.000	0.001	0.010	0.001	0.005	0.001	0.001	0.001
Maximum value	4.470	2.242	2.150	3.800	3.380	2.400	3.800	3.800
Mean	0.176	0.160	0.246	0.203	0.174	0.297	0.355	0.274
Median	0.130	0.080	0.200	0.160	0.140	0.230	0.300	0.220
Geometric Mean	0.107	0.083	0.188	0.148	0.130	0.212	0.282	0.214
Standard Deviation	0.187	0.205	0.187	0.203	0.153	0.244	0.253	0.220
Coefficient of variation	1.06	1.29	0.76	1.00	0.88	0.82	0.71	0.80
Skewness	4.15	2.99	2.17	5.50	1527.61	1.93	2.94	3.61
Kurtosis	42.53	16.86	12.51	58.53	0.88	9.37	20.16	30.44

A boundary analysis was done to examine how the Cu grades change (gradationally or rapidly) across the Haib deposit. A challenge that existed in this exercise was the fact that the majority of the boreholes are close to vertical. This is effectively parallel to the maximum direction of grade continuity as determined from the directional semi-variograms. Boreholes that intersect this trend at a high angle are preferable, nevertheless, some of the Teck drilling was inclined. From these inclined boreholes, holes representing various orientations and dips were selected and the Cu% grade was graphed against depth from collar. These are shown in Figure 14-3.

In Figure 14-3 it can be seen that in every instance there is a well-defined gradual change in grade at the large scale. Minor high-grade inflections do exist but these are highly localised and probably represent the intersection of mineralised veins or other structure. The trend is subtler in TCDH011 primarily because the background grades in this hole are low as it has been drilled on the periphery of the deposit. Though subtle, the pattern is still there and is evident between collar and ~150m depth below collar. In summary; no evidence exists from the boundary analysis of the existence of distinct highly mineralised and less mineralised zones. Grades increase and decrease gradually, with local variations (associated with minor structures).

The above confirms the mineralisation model which states that the mineralisation is spatially associated with syn-mineral dykes (QBP) as well as dark micaceous veins (XPBX). In addition, there is considerable vein development and disseminated mineralisation in the QFP and FP rocks as well. As, the imprint of mineralisation clearly crosses lithological contacts and structures, stationarity (a requirement for estimation) within the lithologies is somewhat lacking.

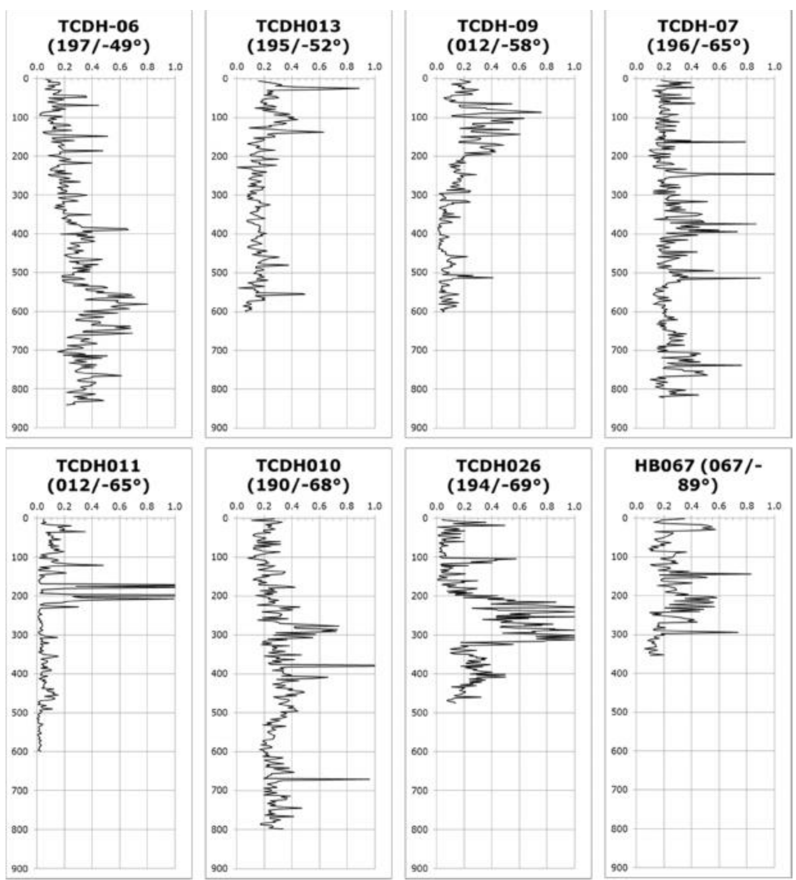


Figure 14-3: Boundary analysis of Cu grade from selected inclined boreholes drilled by Teck.

For the reasons above, the decision was taken to not apply any domaining based on lithology, structure or grade to the definition of the mineral resource for the Haib deposit. Boundaries would be managed based on proximity to data.

14.4 Statistical Analysis

14.4.1 Univariate Statistics

The univariate statistics for Cu and Mo are shown in Figure 14.4. Both populations are positively skewed, particularly the Mo grades. For Cu, the log histogram hints at two possible populations, the first around 0.008% (80ppm) and the second one at 0.18%. It is likely that this lower population represent some non-mineralised portions on the periphery of the Haib deposit. Multiple populations are also indicated for Mo in the log histogram at 5ppm, 10ppm and 20-30ppm.

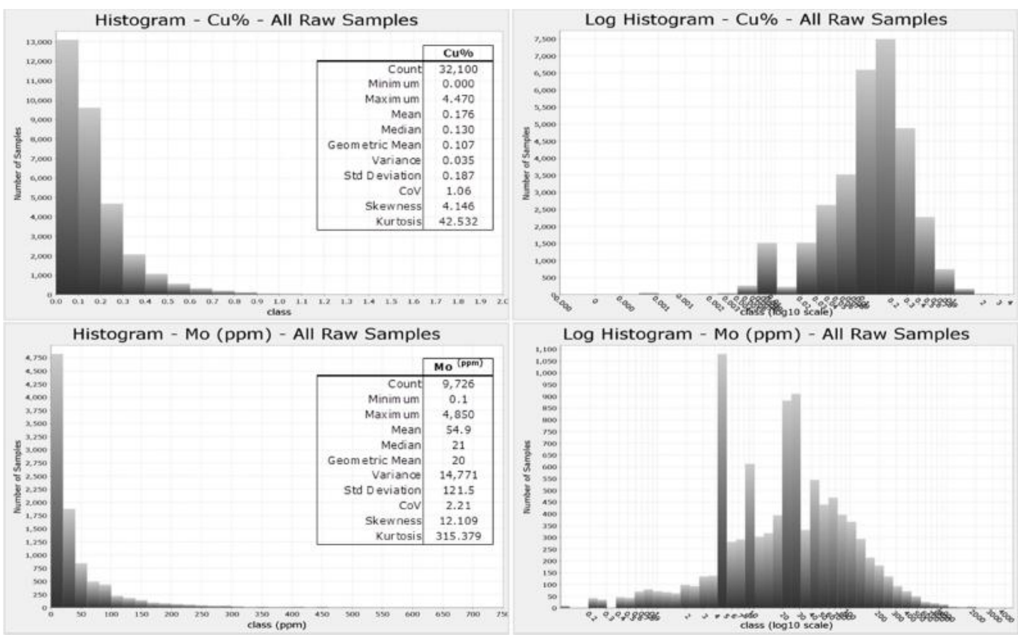


Figure 14-4: Normal and log-histograms as well as summary statistics for all raw Cu and Mo grades.

14.4.2 Grade versus Sample Width

Due to the fact that grade is not strictly additive; the relationship between a sample grade and the width/volume/tonnage it represents is a very important consideration. In some deposits, clear relationships (positive or negative) exist between grade and sample width and in these instances, it is more correct to work with the grade accumulation (grade x width or grade x volume etc.) than the actual grade. In this instance, the core diameter is assumed constant and a default density was to be applied so it made sense to only consider the grade and sample length relationship. A scatterplot of Cu and Mo grade versus sample length is shown in Figure 14-6.

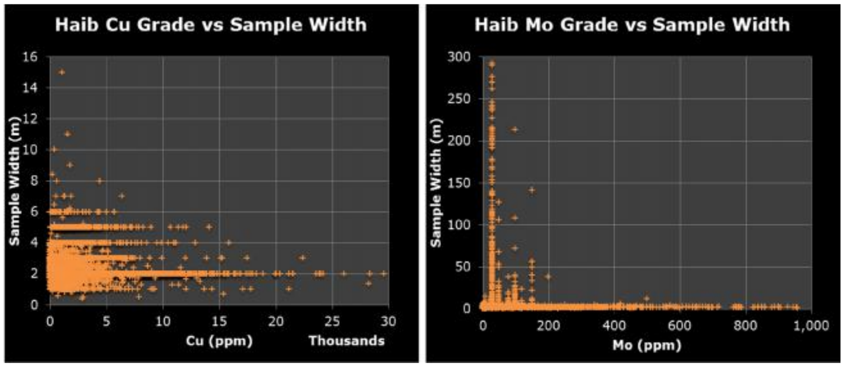


Figure 14-5: Scatterplots for all raw samples comparing Cu and Mo grade to sample width.

It is clear from the figure above that there is no clear relationship between Cu grade and sample width which is confirmed by a correlation co-efficient of 0.013 (calculated but not shown). As the sample size has no obvious effect on the grade, it was decided to continue with the mineral resource estimation work using the Cu grades “as-is” and not accumulations. The same is true for the Mo grades and the same decision to work with Mo grades was taken.

14.4.3 Compositing

Within the portion of the prospecting area that had been modelled by Teck, the horizontal drill hole spacing closely approximates a grid of 150 x 150m. As most of the raw samples are between 1 and 2m wide, the vertical component of the sample spacing is very small relative to the horizontal components. Compositing is typically used to regularise the sample size to produce a standardised weight for each sample. However, in this instance, as the sample lengths are already fairly consistent it was decided to composite the samples to a more global scale better suited to the scale of open cast mining. A 10m composite length was selected to correlate with a typical bench height and 10m composites were calculated starting from the collar. Residual composites were retained.

The univariate statistics were then calculated for Cu% and Mo (ppm) and are presented in Figure 14-6 while the results are tabulated in Table 14-4. From Figure 14-6, it can be seen that the composite populations remain positively skewed. The multiple populations hinted at in the raw samples have been smoothed out in the composites. The Cu composites show a clear compound log normal distribution indicating due consideration must be given to anomalously low and high grades. The Mo shows a log normal distribution around a mean of 30ppm.

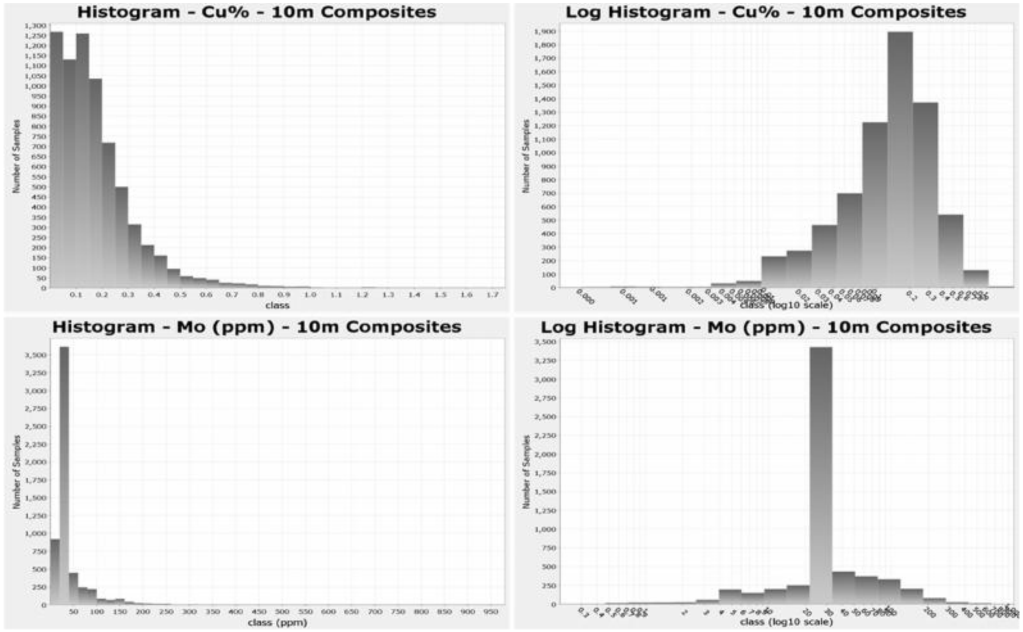
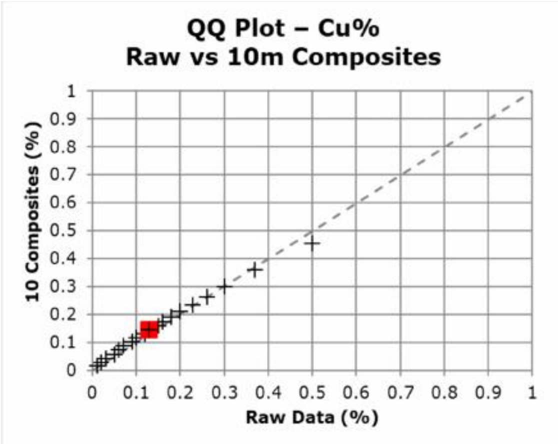


Figure 14-6: Histograms showing the distribution of the 10m composite Cu and Mo grades.

Table 14-4: Univariate statistics of the 10m composites

	Cu %	Mo (ppm)
Count	6,947	5,851
Minimum	0.000	0.2
Maximum	1.748	963
Mean	0.176	43.1
Median	0.144	30
Geometric Mean	0.118	30
Variance	0.0226	2,639
Std Deviation	0.150	51.4
CoV	0.853	1.19
Skewness	2.089	6.099
Kurtosis	10.809	66.107

Smoothing of grade is a natural consequence of compositing and care should be taken to avoid smoothing out all the natural variation of the grade. Creating 10m composites from 1 to 2m samples is quite an aggressive approach so the impact of the compositing was assessed. This was done using Quantile-Quantile (QQ) Plots to compare the percentile distributions of the raw and composited data. These are given in Figure 14-7.



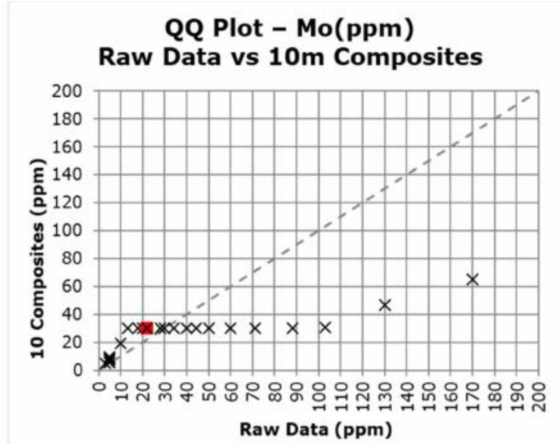
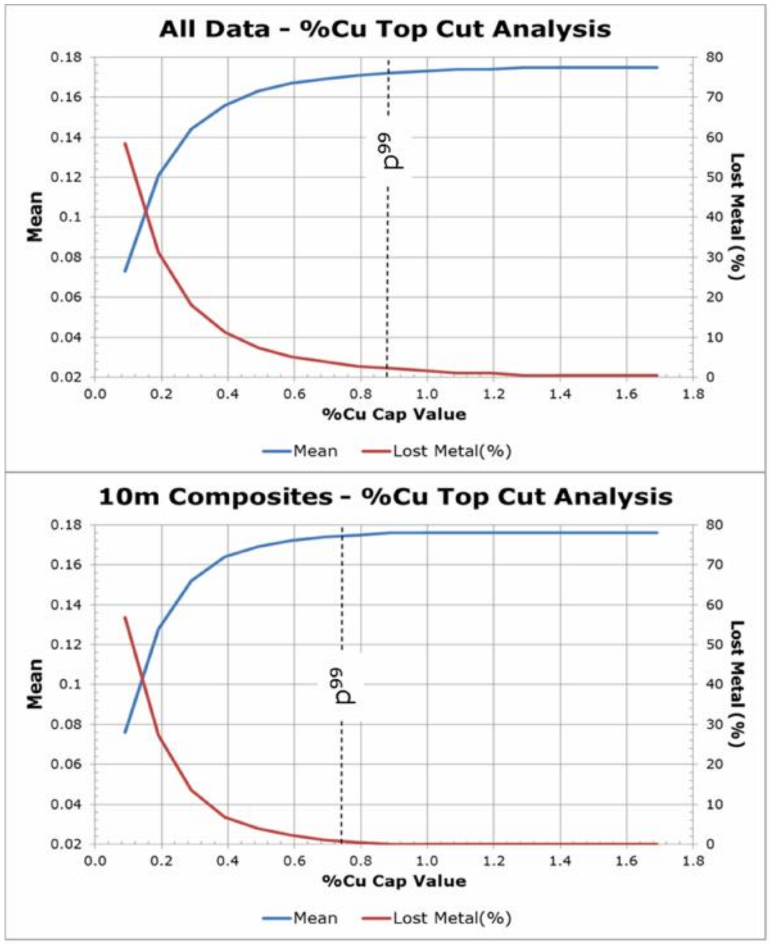


Figure 14-7: QQ Plots comparing the Cu and Mo grade distributions of the raw data against the derived 10m composites.

On a QQ Plot, one expects to see the curve cross the dashed 45° line at the median (red shaded point) or mean value (indication of bias) while the amount of rotation from the 45° line provides an assessment of the amount of smoothing that has occurred. From Figure 14-7, it can be seen that the effect of compositing on the Cu distribution is negligible and the composites reflect a similar variation to the original data. For Mo, the situation is a little more complex but is almost certainly the result of the visually estimated Mo grades discussed further in section 14.4.6. It should also be remembered that Mo is of secondary importance to Cu in this exercise.

14.4.4 Grade Capping

Grade capping analysis was done on the %Cu grades of the raw data as well as the 10m composites. The former in order to test the extent to which capping was required for the compositing while the latter was done to evaluate the risk of extreme assay values during estimation and whether it was necessary to limit their influence during the estimation process. Figure 14-8 shows a Top Cut Analysis for the raw Cu grades as well as the 10m composites. These were derived by applying an upper cap to the data and calculating the mean of the samples below the cap value (blue lines in Figure 14-8). The Lost Metal% effectively quantifies the amount of “metal” lost due to the exclusion of the grades above the cap (red lines in Figure 14-8). The graph provides a means to determine how sensitive the data are to the inclusion or the exclusion of the values above each cap value with the sensitivity being reflected in the gradient of the curve. For instance, between 0.1% and 0.4%, the application of each subsequent cap has a material influence on the resultant “metal”. Above 0.4% this is more subdued.



*Lost metal is (Average – Averaged Capped)/Average * 100 where Average is the average grade of the composited assays before capping and Average Capped is the average grade of the composited assays after capping.

Figure 14-8: %Cu Top Cut Analysis for raw data and 10m composites

From Figure 14-8 it can be seen that;

- The respective curves for each of the raw data and composites are practically identical.
- For the raw data, beyond the 0.4% Cu cap value, there is very little change in the sample subset mean while the Lost Metal% effectively remains constant.
- The same is true for the 10m composites.
- For these reasons it was decided not to apply capping to the calculation of the 10m composites to be used in estimation.
- For the 10m composites, it is clear that the inclusion or exclusion of samples >0.7% will not have a material effect on the resultant contained “metal”. However, it must be remembered that these are 10m composites and therefore they represent a significant volume i.e. to get a high-grade composite requires multiple high-grade samples. As it was felt that these high-grade composites actually carry the requisite support the decision was taken not to cap or cut these values during estimation. However, to avoid spurious smearing of the grades,

their range of influence during estimation was significantly reduced as described further in this text.

14.4.5 Mo – Analysed Grades versus Visual Estimates

To compensate for missing samples, some of the boreholes contain Mo grades that are based purely on analytical laboratory assays while for others the Mo grades are visual estimates. Visual estimates are subjective and their quality is a function of the skill and experience of the responsible geologist and can therefore result in biased datasets. In this instance, the assays represent some 9,324 samples while there are only 404 visual estimates, so the anticipated influence is small. The cumulative frequency distributions are presented in Figure 14-9.

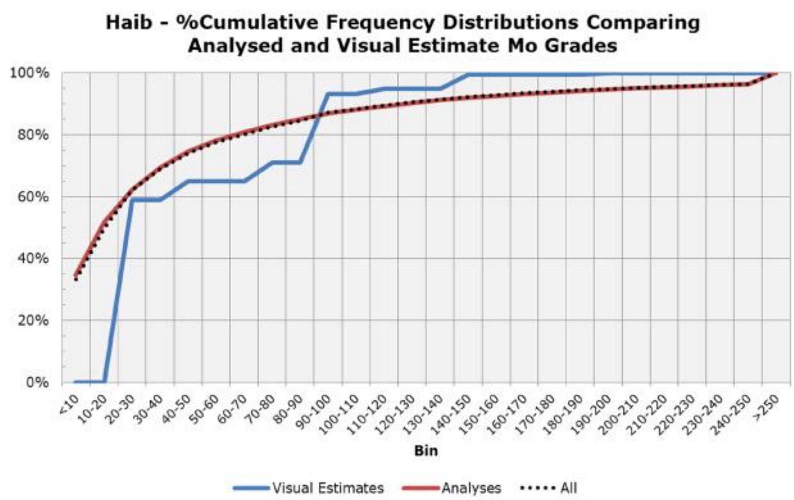


Figure 14-9: Cumulative frequency curves comparing the grade distributions of analysed Mo grades and visual estimates.

From the figure above, the following is evident;

- The distribution of the visual estimates crosses that of the analyses in such a manner that the relatively more high-grade samples are offset by relatively more low grades. This is indicated by the areas between the blue and the red line.
- The distribution of All the Mo grades (visual estimate and analyses) is practically identical to that of the analyses. The visual estimates therefore do not materially bias the final compiled dataset.

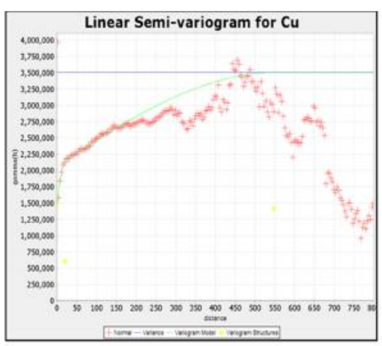
A cautionary note is that the plot above does not consider sample length. The visual estimates are done for significantly larger intervals than the samples. If a 60m visual estimate is composited into six 10m samples, then this will impact the resultant cumulative frequency distribution. The effect of this is clearly demonstrated in the composite distributions in Figure 14-6 above for the Mo plots. While, it is expected that any positive bias will be offset by a negative effect, the Mo grades cannot be viewed at the same level of confidence as the Cu grades and are essentially of secondary consideration in this exercise.

14.5 Variography

In order to detect any preferred directions of grade continuity, variography was conducted for Cu and Mo. This comprised linear semi-variograms to examine the Nugget Effect as well as omni-directional and directional experimental semi-variograms. Anisotropy was determined and variogram models fitted for use in estimation by Ordinary Kriging.

14.5.1 Linear Semi-Variograms

Experimental linear semi-variograms were generated down the hole using the raw data. As linear semi-variograms use the closest spaced samples they can provide a good indication of the degree of randomness (Nugget Effect) of a deposit. The experimental linear semi variograms and the derived variogram models are shown in Figure 14-10. The variograms are very robust and are supported by a large number of sample pairs. Both show double spherical structures with the first Range between 16 and 20m while the Cu shows a Range in excess of 500m. The model curves represent an initial relative rapid change in continuity to the first sill at which point the rate of change is more gradual. For Cu, the Nugget Effect is about 40% of the population variance while for Mo it is at 76%.



	Linear Semi-Variogram	
	Cu	**Mo**
Model Type	*Spherical*	*Spherical*
C_0	1,481,721	11,309
% of Var	42%	76%
$Sill_1$	604,313	1,143
Cum% of Var	60%	84%
$Range_1$	20	16
$Sill_2$	1,417,440	2,381
$Range_2$	548	104

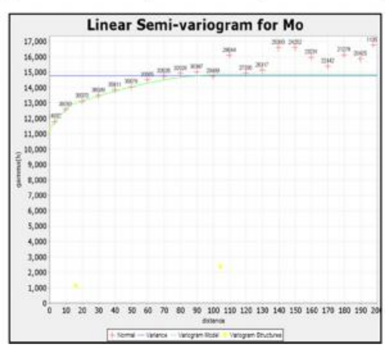


Figure 14-10: Experimental linear semi-variograms for Cu and Mo and the derived spherical models.

14.5.2 Omni-Directional Semi-Variograms

These quantify the rate of change of grade continuity only on the basis of distance without any considerations of anisotropy. Robust spherical variogram models were obtained for both parameters considered and are shown in Figure 14-11. In some instances, outliers were filtered out of the experimental variograms to reduce noise. Models were fitted in this space then back transformed to the original population space. The variogram models are summarised in Table 14-5.

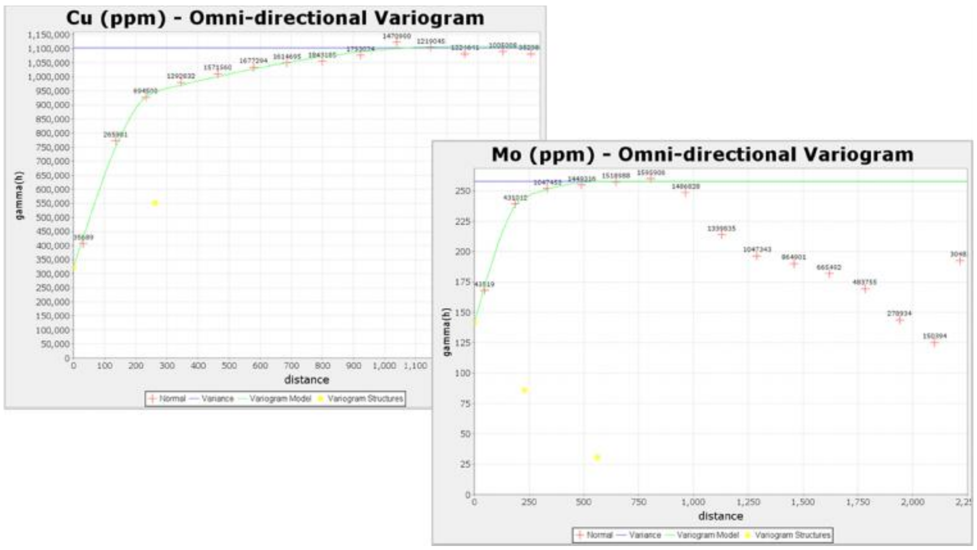


Figure 14-11: Omni-directional semi-variograms for Cu and Mo and the derived spherical models.

Both elements considered could be modelled using a double spherical structure. The largest ranges were obtained for Cu at 1,215m and 560m for Mo. The Nugget for Cu is at ~30% of the population variance while Mo is 55%. All elements reach the first sill at similar levels relative to the population variance (79% - 88%).

Table 14-5: Derived omni-directional variogram models for Cu and Mo by domain

	Cu	Mo
Model Type	Spherical	Spherical
C_0	657,477	1,448
% of Var	29%	55%
$Sill_1$	1,126,183	880
Cum% of Var	79%	88%
$Range_1$	263	230
$Sill_2$	477,570	311
$Range_2$	1,215	560

14.5.3 Directional Semi-Variograms

Although well supported omni-directional variogram models were obtained, one of the drawbacks of omni-directional variograms is that they can often obscure finer scaled details. Additionally, in this instance the mineralisation has a component of structural control with an association with veins and fractures. For these reasons, the presence of any existing potential preferred orientation of grade continuity was tested using directional semi-variograms.

The derived directional semi variogram models are shown in Figure 14-12. The maximum continuity is shown by the red lines while the subordinate anisotropy axes are indicated by the magenta and cyan lines. These show a general anisotropy where the semi-major axis has a Range of ~50% of the major axis and the minor about 25%.

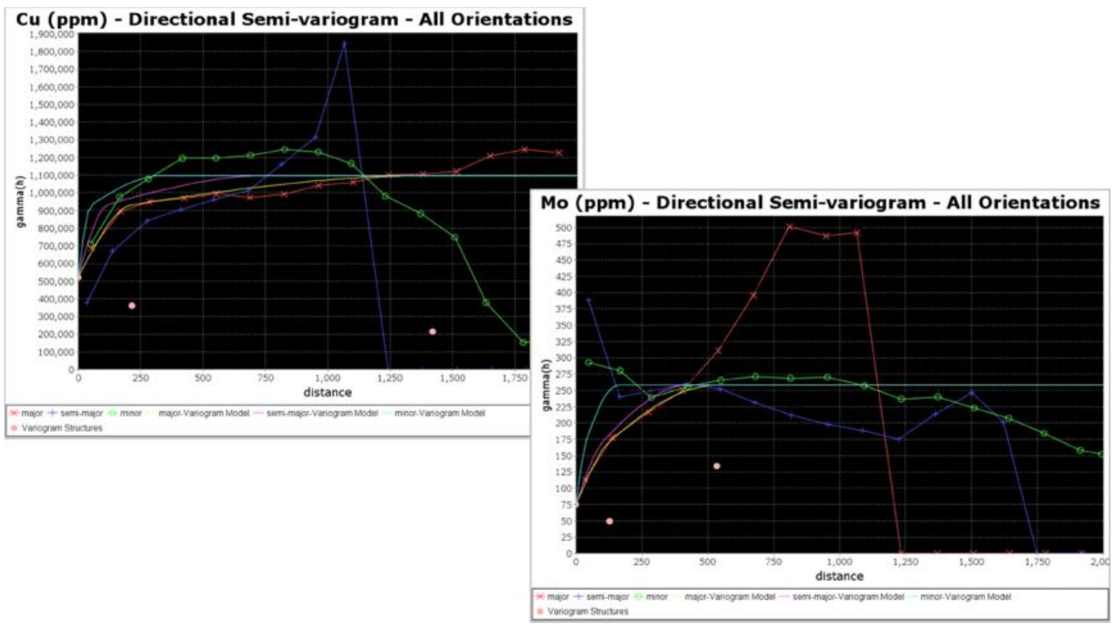


Figure 14-12: Directional semi-variograms showing the “All Orientations” variogram models for 10m composites of Cu and Mo.

The variogram models are summarised in Table 14-6. For Cu, the model shows a double spherical structure with a maximum Range of just over 1,400m. Mo also shows a double spherical model structure with a maximum range just over 530m.

Table 14-6: Derived variogram models and their associated anisotropy components for Cu and Mo

		Cu	Mo
	Model Type	Spherical	Spherical
	C_0	1,073,898	766
	% of Var	47%	29%
	Sill1	744,247	505
	Cum% of Var	80%	48%
	Range1	215	127
	Sill2	443,085	1,369
	Range2	1419.958	534.0
Anisotropy	Plunge	0.0	64.5
	Bearing	134.7	207.1
	Dip	67.2	-77.9
	Major:semi-major	1.87	1.24
	Major:minor	4.15	3.36

At this stage, the decision was made to use the directional variogram models for Cu in Table 14-6 further because they show a larger maximum Range than the omni-directional variograms and the associated anisotropy was felt to be more representative of the Cu mineralisation. For Mo, the omni-directional variograms are more robust and as the directional and omni-directional show very similar maximum ranges, it was decided to use the omni-directional variogram provided above in Table 14-5.

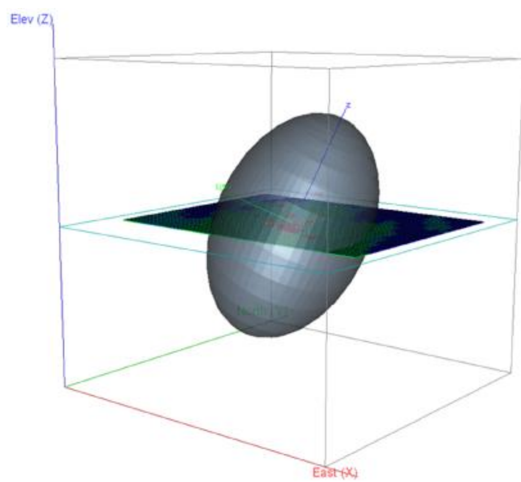


Figure 14-13: Isometric view looking northwest showing the semi-variogram anisotropy for Cu.

The anisotropy ellipsoid for Cu is shown in Figure 14-13. Maximum grade continuity is oriented approximately horizontally along 135°. The semi-major dips towards 225° at about 67°.

14.6 Block Modelling

The general mine planning software, Geovia-GEMS from Dassault Systems was used for this work. GEMS make use of a Percent block model attribute and not sub-celling to manage and report volume accurately. The cell size used in the estimation in GEMS is therefore purely a function of the data spacing. For this work, it was felt that a full Quantitative Kriging Neighbourhood Analysis (QKNA) was un-warranted due to the fact that the drilling is relatively evenly spaced and the derived variogram models are robust and supported by a large number of sample pairs. Any gains from the QKNA are likely to be minor and wouldn't be substantiated by the amount of work required.

Instead, a horizontal cell size of 75m x 75m was used as the drill holes are spaced on a grid with a general spacing of 150m x 150m. A cell height of 10m was selected.

The block model project was positioned over the main target area on which the 150m x 150m pattern had been drilled and sized appropriately to cover the full extent of the drilling with depth. The geometrical definitions are given in Table 14-7.

Table 14-7: Block model geometrical definitions

Origin	X	780,140
	Y	6,821,810
	Z	650
Cell Size	Column	75
	Row	75
	Level	10
No. of Cells	Columns	39
	Rows	31
	Level	115
Length	X Direction	2,925
	Y Direction	2,325
	Z Direction	1,150

14.7 Specific Gravity

Limited specific gravity (SG) determinations were done by Teck on core from boreholes TCDH014 up to and including TCDH027 (excluding TCDH025). A total of 99 SG determinations were available for this work. Figure 14-14 shows a normal histogram of these along with the summary statistics. The distribution is quite normal with the mean and median values being identical at 2.76 T/m^3. Values range between a low of 2.23 and a high of 3.01 T/m^3.

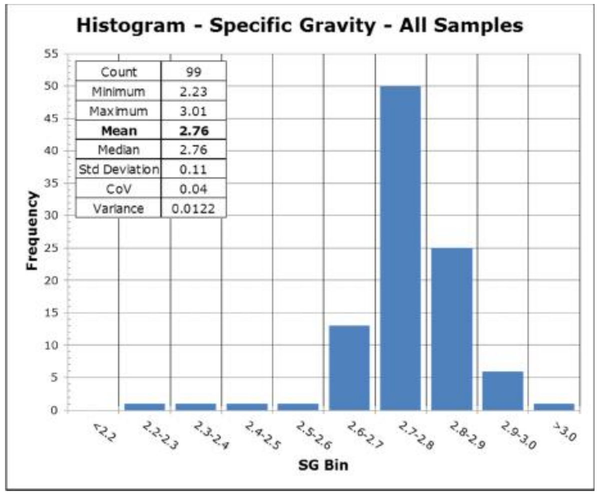


Figure 14-14: Normal histogram of available specific gravity determinations

As there were not enough determinations to allow for interpolation of the SG for the Haib deposit, it was decided to apply a default SG of 2.8 tonnes/m^3 to the Haib mineral resources. This figure was derived by rounding the mean SG of 2.76 to one decimal place. It is the author's opinion that this is a perfectly reasonable approach considering the low variability of the SG's determined for the Haib deposit.

14.8 Estimation

Ordinary kriging was used to estimate Cu and Mo in a stepwise fashion as follows;

- A first pass kriging run was done using a search ellipse matched to the Ranges of the semi-variogram models. Cells estimated during the first pass were tagged with an integer value of 1.
- The search ellipsoid Ranges were then doubled, the minimum and maximum number of samples adjusted and a 2nd kriging run was done. Cells populated were tagged with the value 2. In both instances, all cells were estimated within a search of 2x the Range.
- An ellipsoidal search was used for both Cu and Mo.
- No high-grade limits were used. Instead a high-grade transition (HGT) value for Cu was defined at 1%. Cu grades above this value were used "as-is" but the range of influence for these was reduced significantly.

A summary of the kriging run inputs is given in Table 14-8.

Table 14-8: Summary of the kriging inputs for each of the runs completed for Cu and Mo.

		Cu	Mo
Run 1	Search	RANGE	RANGE
	Minimum Samples	10	10
	Maximum Samples	18	18
	Discretisation	10x10x5	10x10x5
	Search Type	Ellipsoidal	Ellipsoidal
	High Grade Transition (HGT)	1%	128
	Range for >HGT	700; 380; 170	50; 50; 50
	No. of Cells Estimated	108,758	90,229
	Estimates Cells as %	>99%	83%
Run 2	Search	RANGEx2	RANGEx2
	Minimum Samples	12	16
	Maximum Samples	24	20
	Discretisation	10x10x5	10x10x5
	Search Type	Ellipsoidal	Ellipsoidal
	High Grade Transition (HGT)	1%	128
	Range for >HGT	700; 380; 170	50; 50; 50
	No. of Cells Estimated	108,758	18,427
	Estimates Cells as %	<1%	17%

During kriging, various outputs such as kriging variance, kriging efficiency, slope of regression, number of samples, number of negative weights and others were tracked and used as a guide in the estimation process.

14.9 Estimate Validation

During kriging, various parameters were tracked and trace blocks were used in regions of high, medium and low data support. Post-estimation, visual inspection was used along with more quantitative methods such as;

- Non-spatial comparison of source data and estimates using QQ Plots
- Swath lots were generated to compare trends in the data with estimates
- Comparison of estimates and average grades of informed cells as well as estimation methodologies

14.9.1 QQ Plots

The Quantile-Quantile Plots comparing the percentile distributions of the 10m composite source data to the estimates are shown in Figure 14-15. Smoothing is a consequence of estimation and as expected is reflected by the rotation of the curves from the dashed 45° line. For Cu the estimate distribution is lower grade than that of the source data. This is probably due to the fact that the domain extends beyond the data limits (particularly with depth). In these areas, grades are lower but also a lot of cells will be populated by only a portion of the source data. As the QQ Plot is non-spatial, it cannot really account for this. Nevertheless, the fact that the estimates are slightly lower grade does imply a more conservative result and it is the opinion of the author that these results are a reasonable representation of the source data.

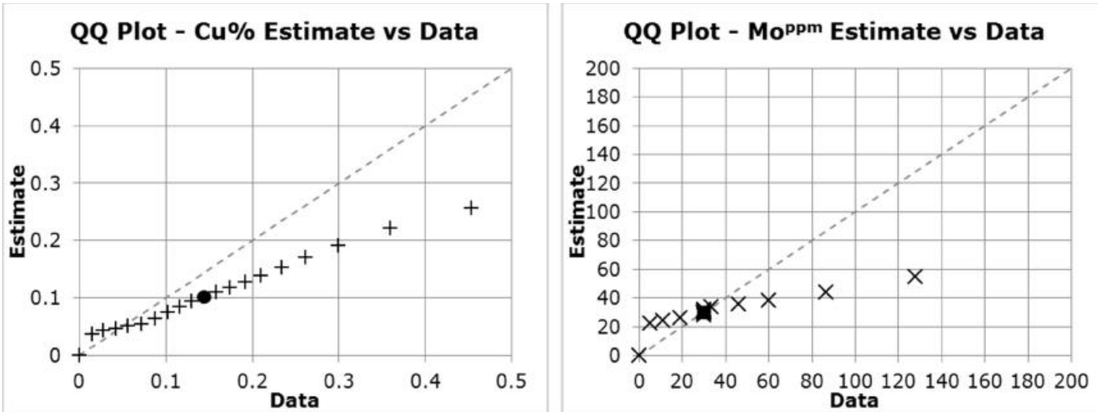


Figure 14-15: QQ Plots comparing the grade distributions of the source composite data and the estimates.

14.9.2 Swath Plots

Swath plots involve the aggregation and calculation of average grades of samples and estimates along pre-defined corridors orientated along the X, Y and Z axes of the block model. As they are aggregations, they are used to test whether data trends are reflected in the estimates e.g. Areas with high grade samples are associated with high grade estimate values. The generated swath plots for Cu are shown in Figure 14-15 where it can be clearly seen that the estimate and data trends show good correlation.

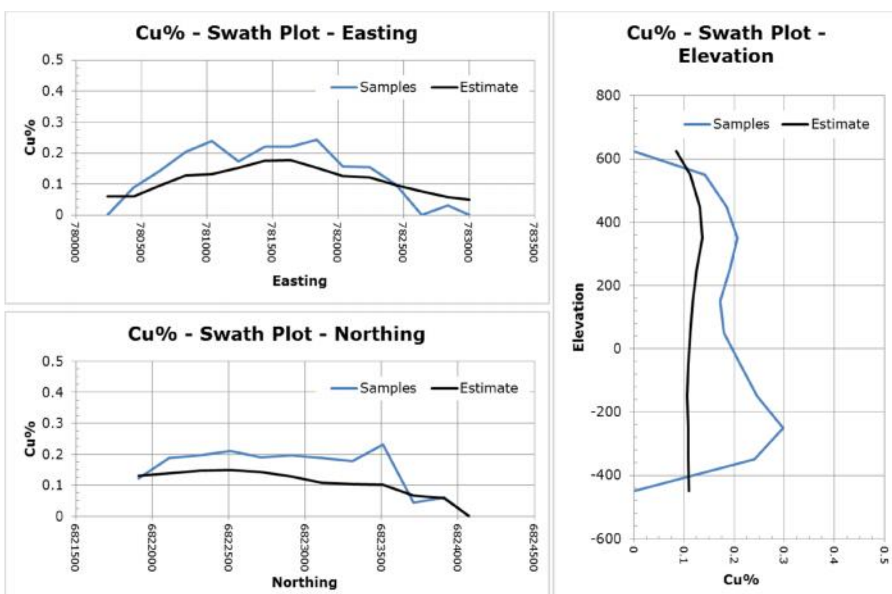


Figure 14-16: Swath plots showing trends in source 10m composites and the estimates of Cu.

14.9.3 Comparison of Estimates, Informed Cells and Methodology

An inverse distance squared estimate ("ID^2") was prepared for the blocks using the same search criteria used for ordinary kriging ("OK") and the results were compared to the OK estimate. In addition, the average composite sample grades for all blocks containing composite samples (informed blocks) were compared to the OK and ID^2 estimates using scatter plots. These scatter plots are shown in Figures 14-17.

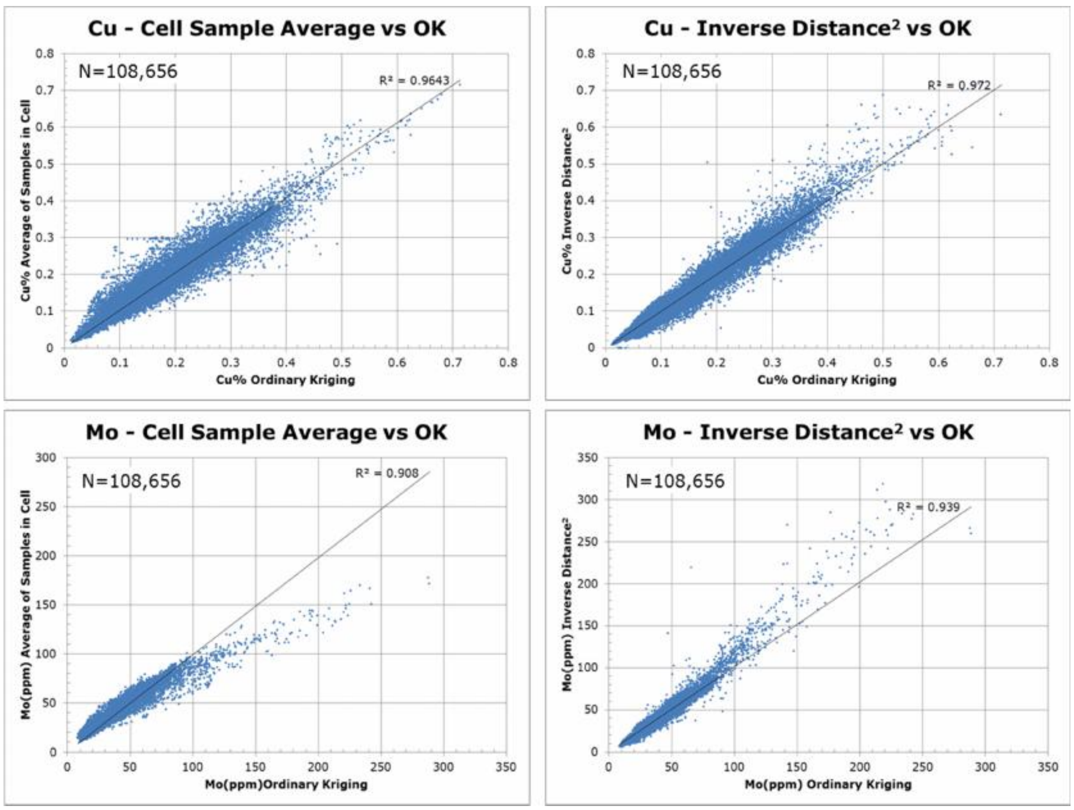


Figure 14-17: Scatterplots comparing Ordinary Kriging estimates to Inverse Distance^2 and Ordinary Kriging estimates to informed cell averages

In Figure 14-17 it can be seen that for Cu there is good correlation between the OK estimates and average grades of informed cells (R^2 = 0.964). The same is true for the OK and ID^2 estimates as reflected in the R^2 of 0.972.

For Mo grades above 80ppm (~p90) the OK estimates are higher than the average grade of the informed cells while relative to the ID^2 estimates, the OK results are lower at Mo grades >80ppm. The reasons for this are not fully understood however with R^2 values exceeding >0.9 the correlations are still reasonable.

14.10 Block Model Sensitivity Analysis

The tonnage and grades of the Haib deposit calculated here are a direct function of the cut-off grade used for reporting. To illustrate this, the block model quantities and grade estimates within the total block model are presented in Table 14.9 at different cut-off grades. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade and the associated grades and tonnages.

Table 14-9: Block model quantities and grades at the specified Cu% cut-offs

Cu% Cut-off	Million Tonnes	Cu%	Mo(ppm)
≥0.60	2.4	0.64	55
≥0.55	5.0	0.60	49
≥0.50	9.1	0.57	53
≥0.45	20.4	0.51	56
≥0.40	46.7	0.46	63
≥0.35	137.0	0.40	59
≥0.30	334.9	0.35	55
≥0.25	1,067.2	0.29	53
≥0.20	2,196.6	0.26	48
≥0.15	4,156.1	0.22	42
≥0.10	8,198.0	0.17	37
Total	15,736.0	0.12	33

* The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

From Table 14-9 it can be seen that as the Cu% cut-off is increased in increments of 0.05% from 0.10% to 0.25%, the tonnage is effectively halved. Between 0.25% and 0.30% the tonnage is reduced by two thirds and this rate continues until 0.40% after which the rate of drop returns to a halving. The grade tonnage curve associated with Table 14-9 is shown in Figure 14-18.

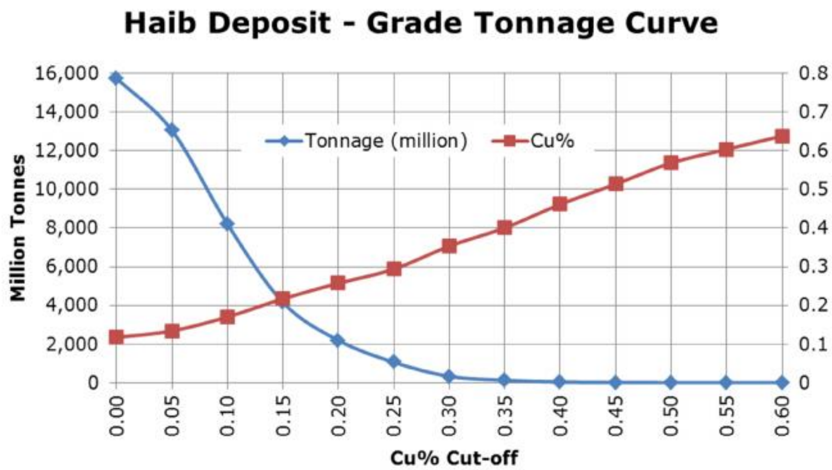


Figure 14-18: Grade tonnage curve for the Haib deposit.

14.11 Mineral Resource Classifications

The mineral resource estimates presented here have been classified according to the guidelines of the Canadian National Instrument 43-101 by Dean Richards of Obsidian Consulting Services, who is an appropriate Qualified Person as defined by the instrument. The definitions applied from the code were as follows;

Mineral Resource

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.
Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

The types of data, data density and the data density for Haib are such that they provide a good basis for the confident interpretation of the geology and mineralisation constraints of the Haib deposit. The drill hole spacing and the quantity of data has allowed the grade continuity to be well defined at distances much smaller than the Ranges expressed by the variography. While a significant portion of the data was not subjected to an international standard Quality Assurance and Quality Control programme, the most recent work completed by Teck was significant and as it sampled largely the same domain as the historical work, it provides a means of establishing the quality of the historic data. These show that the Cu distributions of the historic data and that of Teck are practically identical.

With respect to the Mo grades reported here, they do not provide the confidence levels that the Cu grades do and the resource classification is based purely on the Cu grade.

Under the considerations above, the following classification has been applied. No Measured Mineral Resources can be declared for Haib at this stage. From topography to an elevation of 75m above mean sea level (amsl), the data density across most of the estimated resource is high being associated with a drill grid approximating 150m x 150m. This spacing is substantially smaller than Ranges obtained for Cu both from linear and directional semi-variograms and thus the portion of the mineral resource supported by this data density to an elevation of 75m amsl is classified as Indicated Mineral Resources.

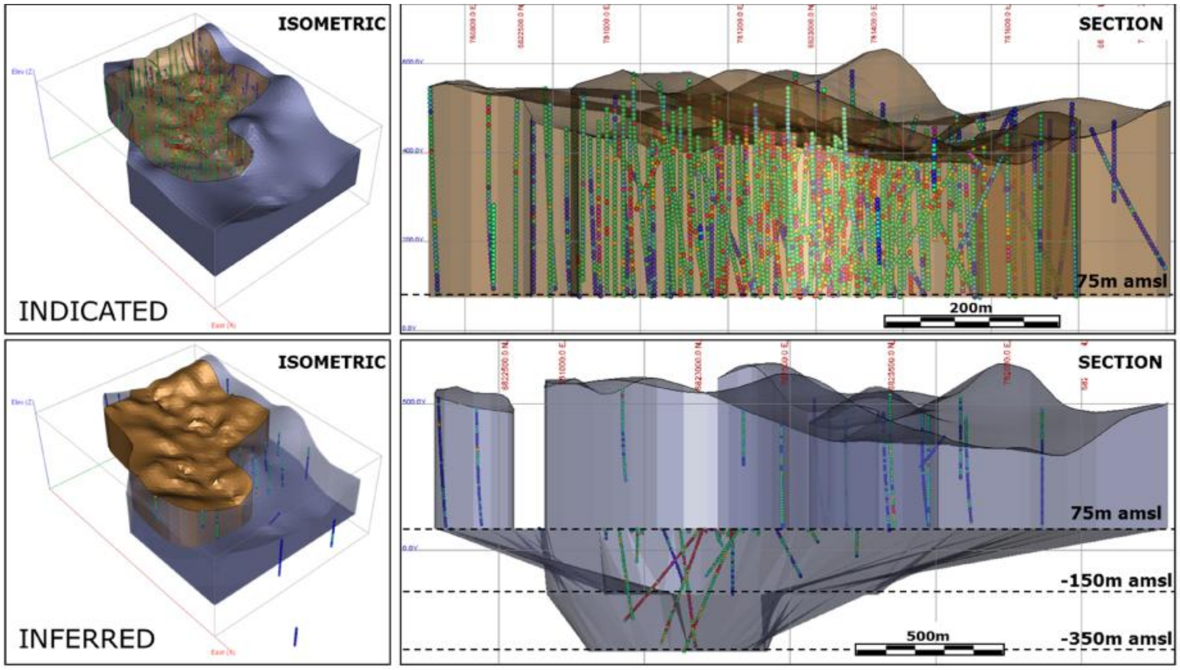


Figure 14-19: Isometric and sectional views showing the distribution of drill hole data and the applied mineral resource classification.

On the periphery of the 150m x 150m grid, along the northern and eastern edge of the deposit, the data density increases to a grid approximating 300m x 150m with holes drilled to 75m amsl. This portion has been classified as an Inferred Mineral Resource and extends eastwards to the edge of the estimated volume. This extension is supported by the fact that there is a line of boreholes in this area (Inferred isometric view in Figure 14-19) that are spaced within the maximum variogram Range Cu.

Below 75m amsl, the number of holes drilled deeper decreases rapidly. 11 boreholes are drilled to an elevation of -150m amsl while 6 of these are drilled even deeper to -350m amsl (Figure 14-19). Using an area of influence of 250m approximating the short Range spherical structure seen in the Cu variogram, areas of influence were created around these data points at -150m amsl and -350m amsl. The resulting volume is classified as Inferred Mineral Resources.

The fact that data density has been used primarily as the basis for the mineral resource classification is clearly illustrated in Figure 14-19 above. As a supplementary exercise, the Slope of Regression and Kriging Efficiency parameters generated for Cu during the ordinary kriging were cross referenced to the mineral resource classification.

This showed that within the Indicated Mineral Resources, the average Slope of Regression is 0.85 (target 1) while the Kriging Efficiencies are positive and average 76% (target +100%). For the Inferred Mineral Resources, the average Slope of Regression is 0.56 (target 1) while the Kriging Efficiencies are positive and average 46% (target +100%). Although not the basis for the Mineral Resource Classification, they do confirm that the classification applied is reasonable. Furthermore, it is the author's opinion that the Indicated Mineral Resources are suitable for use in a Pre-Feasibility level study.

14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

"A concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In our (Peter Walker and Dean Richards) opinion, and assuming the current copper prices are maintained, there is a reasonable prospect for economic extraction and we therefore assume that at a cut-off grade of 0.25% all the processing options will show positive economic results and for this reason, a cut-off grade of ≥0.25% Cu has been applied in the compilation of the Haib Mineral Resources Statement presented in Table 14-10. It must be highlighted again that the confidence in the reported Mo grades is significantly lower than Cu and only the Cu grades have been used in the classification. The Mo grades reported in Table 14-10 are provided for illustrative reasons only.

Table 14-10: In situ classified mineral resources of the Haib Project at a 0.25% Cu cut-off grade.

Resource Class	Volume (xMillion m^3)	Density	xMillion Tonnes	Cu(%)
Measured	-	-	-	-
Indicated	163.2	2.8	456.9	0.31
M+I	163.2	2.8	456.9	0.31
Inferred	122.3	2.8	342.4	0.29
Rounding has been applied as appropriate to reflect limits of precision and accuracy				

14.13 Areas of Uncertainty of Resource Estimates

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- A change in Cut-off grades due to changes in Cu pricing and other economic factors.

- Drill spacing - The drill spacing and depth of drillholes at the Haib deposit is insufficient to determine the full extent of the mineralization. The drill spacing proves to be too large to accurately represent grade variations at the smaller scales required for short term planning and scheduling of mining operations.

- The Mo grades are largely based on estimates during logging and not on assay data, so little reliance should be put on the quoted Mo resources.

- If the SG's of the deposit display significantly greater variation than indicated by the limited data set of SG determinations, there is a risk that the conversion of Volume to Tonnage may be inaccurate.

- Commodity prices and exchange rates may materially affect project feasibility.

- Assumptions used in the PEA to generate the data for consideration of feasibility of mining the Haib deposit, such as mining costs, both Capital and Operating, extraction and recovery of Cu and Mo, infrastructure development costs, availability of water, electric power and rail transport.

- Unusual weather phenomena, government regulation changes, governmental failure to provide adequate infrastructure may have material effects on the feasibility of the project.

- The project will require additional approvals, permits and licences in order to begin mining operations and these may be delayed or not granted.

- The Haib project will require substantial additional financing in the future and HM cannot be certain that such financing will be available.

- Any revocation, dispute or challenge to HM's mineral title may have a severe effect on project feasibility.

- Labour disruptions and increased labour costs could have a material effect on feasibility of the project.

- Namibia is a democratic, capitalist country. However, there is a political risk factor which could affect the feasibility of the project and the enforceability of HM's rights. The Fraser Institute Survey of Mining

Jurisdictions in 2016 [53] reported Namibia's score and their rank deteriorated for the second straight year. In 2014, Namibia was ranked as the 19th most attractive jurisdiction in the world when only policies were considered. The country fell to 29th in 2015 and dropped again to rank 38th in 2016. After this decline, Namibia no longer ranks as the second most attractive jurisdiction in Africa based on policy. In 2016 miners expressed increased concern over uncertainty regarding the administration, interpretation, or enforcement of existing regulations (+28 points), the taxation regime (+21 points), and trade barriers (+19 points). Namibia now ranks 9th out of 18 African mining jurisdictions.

15. MINERAL RESERVE ESTIMATE

There is no NI 43-101 compliant reserve estimation for the Haib mineralisation.

16. MINING METHODS

16.1 Introduction

Considering the Haib copper deposit characteristics, the suitable mine design is based on an open pit method. As the deposit is basically composed by hard rock material, the mining operations will involve drill and blast of all excavated material, which will be segregated by cut-off grade.

The mining fleet considered being suitable for the Haib project would most likely consist of between 80 t and 120 t sized hydraulic excavators, off highway dump trucks with a capacity of between 65 t to 90 t, supported by standard open-cut drilling and auxiliary equipment.

16.2 Geotechnical Review

16.2.1 Pit Slope Assessment

A further geotechnical evaluation of the Haib copper deposit is required as it is an integral component of any proposed mine or mining project. Parameters established by geotechnical study are fundamental for strategic mine planning and effective technical guidance of the mining operations.

16.2.2 Excavation Characteristics

In terms of the excavation characteristics, even though there is no geotechnical study of the deposit, the current information indicates that drilling and blasting is needed for all excavated material.

16.3 Groundwater Investigation

There is no detailed ground water study of the Haib copper deposit area. The Orange River flows about 15 kilometres south of the main deposit, located in the extreme south of Namibia, where the average annual rainfall is 25-50 mm.

16.4 Proposed Mining Operation

16.4.1 Introduction

Initial analysis involved mining approximately 8.5 Mtpa of ore. There is no estimate of the amount of waste that will be mined in the project.

16.4.2 Open Pit Work Roster

It is suggested to the mining operations to work 365 days in a year, less unscheduled delays such as high rainfall events which may cause mining operations to be temporarily suspended, which is high unlikely to happen considering that average annual rainfall is extremely low, especially in the deposit area.

There are numerous types of rosters, but it can be suggested one in which the mine workforce will operate on a two shift, three panel roster, seven days a week, in two 11 hour working shifts with the equipment services scheduled as required.

For example, a six and three (6/3) roster could be considered, which would equate to 6 days on day shift, 3 days off, 6 days on night shift, 3 days off.

The crushing plant is assumed to operate continuously except for planned maintenance periods.

16.4.3 Bench Design

The height of the mining benches is usually determined according to physical characteristics of the mineralisation and its impact on selectivity and dilution control. Both mineralised material and waste could be drilled and blasted on standard 5m benches for primary crusher feed and possibly 10m benches for waste, and then mined by hydraulic backhoe configured excavators; nominally ranging from two 3m high faces to three 4m high faces, taking into account blast induced swell, into rear dump and off highway haul trucks. The number of flitches to mine a bench will be dependent on the selectivity required and the size of the excavator used.

16.4.4 Drill and Blast

Rock fragmentation will be undertaken by drilling and blasting and its parameters will be based on the rock characteristics obtained during the geotechnical investigation, which will provide information of weathered and fresh material.

The blast pattern is dictated by the powder factor required to ensure appropriate fragmentation and heave. The selection of the powder factor will be based on the UCS (Unconfined Compressive Strength) measurement results obtained from the preliminary excavation characterisation work.

16.4.5 Load and Haul

There is no estimative yet of the total material movement at the project. However, considering the amount of ROM to be processed it is likely to be proposed by contract miners 120 t excavators and a combination of 65 t and 90 t off-highway dump trucks.

The high grade ore will be transported by trucks to the run-of-mine (ROM) stockpile, which will be near by the primary crusher. The distance between the pit and the plant will be established considering further topographic studies and the final mine pit design.

16.4.6 Stockpiling and Reclaiming

It is suggested that the material which does not match with the quality standard grade and is unable to be directly dumped into the crushing circuit be placed in an appropriate stockpile for processing at a later time if it is profitable.

The ROM will be stockpiled directly adjacent to the primary crusher and re-handled with a wheel loader that will dump material into a ROM bin, which feeds the gyratory cone crusher.

16.4.7 Pit Dewatering and Drainage

In the extreme south of Namibia, in summer the rainfall is associated with occasional thunder storms and is of short duration, but can be of very high intensity. Due to this, engineered surface water management structures are suggested to minimize effects of storm water run-on to critical mine facilities and to control the release of mine-impacted water to the environment.

16.5 Contract Mining

It is generally not economic for a mine operator to undertake all of the functions required in the development and operation of a mine. Contractors are usually engaged when funds are not available for equipment purchase, the duration of the task is short, specialist skills are required and/or specialised equipment is involved.

Contractors can be effectively utilised to overcome unavoidable peaks in production required to maintain the mining schedule. For example, an open cut may have a large volume of pre-strip required which can be effectively moved by scrapers before the commencement of a hard rock mining. It is unlikely that the purchase of a fleet of scrapers could be justified to undertake this work which would probably be completed by a contractor in 3-6 months.

Therefore, it is suggested to adopt contract mining instead of an own mining operation. The infrastructure necessary to the mining contractor, such as administration facilities and workshop may be contemplated in the contract as contractor’s responsibility, which will decrease the project’s CAPEX.

16.6 Contract Drilling and Blasting

Considering the same arguments from mining contracts and also for security and quality service reasons, it is suggested to adopt drill and blast contract instead of own operation.

All explosives and accessories must be stored at the planned magazine site and explosive storage facility site. The amount of explosive consumed per week will be defined basing on powder factor (kg/m3 or kg/tonne) and the amount of material mined (ore + waste). As the Haib deposit is situated in a remote area, it is suggested to have explosive storage to operate for a reasonable time.

The explosive storage facility may be contemplated in the contract as contractor’s responsibility, which will decrease the project’s CAPEX.

16.7 Pit Optimisation

16.7.1 Optimisation Methodology

To do pit optimisation the use of mining software is necessary. For a given resource model, cost, recovery and slope data, the software calculates a series of incremental pit shells in which each shell is an optimum for a slightly higher commodity price factor.

The sequence of the pit shell increments is sorted from the economically best (the inner smallest shell viable for the lowest commodity price) to the economically worst (the outer largest pit shell viable for the highest commodity price).

In a pit optimisation, the software provides indicative discounted cashflows for two mining sequences called “best case” and “worst case” scenarios, both using time discounting of cashflows. In the best case, the optimum pit shells are mined bench by bench in increments from inner to the outer shell, resulting in a higher discounted cashflow (DCF) due to lower stripping ratios and/or higher grades in the early years of mine life. The worst case scenario is based on mining the whole pit outline bench by bench as a single pit, hence resulting in a lower DCF as a result of usually high stripping requirements in the early years of the operation.

Ordinarily, after the selection of the ultimate pit, several practical mining stages are designed and sequenced when developing a final production schedule. This sequence would provide a discounted cashflow somewhere between worst and best case scenarios. For this reason, the average discounted cashflows are calculated for each pit shell (mean of the worst and best cases) in order to emulate a practical mining sequence.

The cashflows, as described above, are exclusive of any capital expenditure or Project start-up costs and should be used for pit optimisation comparison purposes only. No project Net Present Value (NPV) can be derived from these cashflows.

16.7.2 Overall Pit Slopes

The overall pit wall slope angle, which is essential for the pit optimisation study, must be based on the geotechnical parameters established by further geotechnical study.

16.8 Mine Design

The mining design will be determined considering economics, engineering and geological structure aspects. In terms of geological aspect, a further investigation will be necessary to establish the parameters and create a detailed block model, which will be based on geostatistics and the geological data gathered through drilling of the prospective ore zone.

16.9 Tailing Disposal

16.9.1 Introduction

Option 1 and 2 will generate approximately 250 thousand tonnes per annum ("ktpa") of tailings from the flotation circuit. Due to environmental and water recovery considerations the tailings will undergo dry stacking. All options include dry stacking of the iron oxide waste from the iron removal stage (250-500 ktpa depending on the process option). The remaining waste will either be from the ore sorter rejects or from the heap leach pads (~8 Mtpa) and will be coarse rock material. The heaps will remain in place and undergo periodic washing to ensure copper extraction is maximised. Washing will be stopped once the ore is considered 'spent'. The ore sorter rejects and the spent ore can be disposed of in a manner that produces a suitably stable landform.

16.9.2 Environmental

In terms of environmental aspects, dry stack facilities offer a number of advantages to other surface tailings storage options – some of these include:

- Reduced water requirements, principally achieved by recycling process water and near elimination of water losses through seepage and/or evaporation;
- Groundwater contamination through seepage is virtually eliminated;
- Significant safety improvement with the risk of catastrophic dam failure and tailings runout being eliminated;
- Easier to close and rehabilitate.

16.9.3 Tailing Disposal Design

The strength, moisture retention and hydraulic conductivity characteristics of the tailings need to be established for any given project considering the technology. The strength and hydraulic parameters from saturated tailings should be determined to "anchor" the results and tests as variable moisture contents are required to demonstrate the impact of the inevitable range of operating products.

The other important geotechnical characteristic to determine is the moisture-density nature of the tailings. The unsaturated moisture-density relationship indicates in-situ density expectation as well as the sensitivity of the available degree of compaction for any given moisture content. From a compaction perspective, the filtered tailings should neither be too moist nor too dry. The optimal degree of saturation is usually between 60 and 80%.

16.10 Waste Rock Storage

16.10.1 Introduction

It is suggested to consider stockpiling the low-grade ore to process it at the end of mine life, in case the copper price increases considerably by the end of the mine life and/or a new mineral processing technology is created or developed.

16.10.2 Waste Rock Storage Design

The overall rock storage design is dependent on a number of factors, such as:

- Topography of the dump site;
- Method of construction;
- Geo-technical parameters of mine waste; and
- Geo-technical parameters of the foundation materials.

All of these factors combine in various ways during the life of a mine waste dump to aid in the stability of the dump or to contribute to its instability.

17. RECOVERY METHODS

For the recovery of copper from the Haib deposit, heap leaching was considered for all options. The primary reasons for the selection of heap leaching are the low grade nature of the deposit and the vast scale of the orebody. Previous work conducted on the Haib project suggests that a conventional crush-grind-float and sale of copper concentrate is not economically feasible due to the low grade and hardness of the ore – requiring a significant amount of energy for grinding. The low costs associated with heap leaching compared to a whole ore flotation circuit are believed to improve the viability of the project. Heap leaching is traditionally performed on oxide material, although there has been increasing development in the application to acid insoluble sulphides. Previous sighter amenability testwork suggests the Haib material can extract high amounts of copper, up to 95.2% via a bacterial assisted leaching, although additional testwork is required to determine the optimal operating parameters. Given these results there is no reason to suggest the chalcopyrite in the Haib deposit will not be amenable to bacterial assisted heap leaching.

The initial flowsheet development was based on a high grade/waste reserve scenario, which involved a 0.25% copper cut-off grade giving 167.5 Mt at 0.3271% copper and 56 ppm molybdenum. A project life of 20 years was selected as the basis going forward, which corresponded to a throughput of approximately 8.5 Mtpa. The flowsheet and subsequent mass balance, equipment sizing and capital estimate calculations were performed using this 8.5 Mtpa base case. Each document and calculation was then updated to reflect the measured and indicated resource of 456.9 Mt at 0.31% copper and an expected 46 ppm molybdenum (not currently indicated). At the same annual throughput, this would correspond to a 55 year project life. Due to the unrealistically long project life, it was suggested to start at 8.5 Mtpa and operate at this throughput for approximately 3 years and then execute staged expansions to eventually ramp up to 20 Mtpa, ultimately shortening the project life. As the resource expands and the inferred data progresses towards measured, then additional expansion to possibly 40+ Mtpa should be assessed. All flowsheets, mass balances, design criteria and equipment lists are based on an 8.5 Mtpa throughput; although financial components have been scaled for the higher throughput scenario (see section 22).

Four options were established for the purposes of the economic evaluation:

- Option 1: Ore sorting upgrading, dense media upgrading, flotation and heap leaching of the tails
- Option 2: Two-stage dense media upgrading, flotation and heap leaching of the tails
- Option 3: Ore sorting upgrading and heap leaching of the upgraded material
- Option 4: Whole ore heap leaching

All options include molybdenum recovery, although the operating costs, capital costs and revenue have not been included, and will be considered if the molybdenum progresses into indicated resource.

17.1 Ore Transport

The Haib copper deposit it situated in highly undulating terrain. Heap leaching using a valley heap method would be suitable considering the topography, although the cost of associated earthworks to provide a flat surface for the process plant and the cost associated with transportation of raw material in and products out warrant the placement of the process plant on flatter ground. A long distance conveyor (4.5-5 km) has been proposed, which would transport crushed ore from the mine site to the process plant for subsequent grinding. The long distance conveyor cost has been included in all options.

17.2 Process Description – Option 1

17.2.1 Crushing and Ore Handling

Run of Mine (ROM) ore is transported by truck from the mine to the ROM stockpile area near the crushing plant. The material is transferred to a ROM bin, which feeds to a primary crusher. The primary crusher is a gyratory crusher suited to higher crushing capacities. The open side setting (OSS) is expected to be set at 127 mm with an assumed P_{80} of 127 mm to be produced. The output of the gyratory crusher is fed to a primary crusher screen with an aperture of 20 mm. The oversize material feeds a group of diverter chute and conveyors that evenly distribute the material into four streams. Each stream feeds the ore sorting circuit which consists of four ore sorters arranged in parallel. The material rejected by the ore sorters feeds the reject stockpile via a conveyor.

The ore sorter ‘accept’ products go to a conveyor and then to a diverter chute that evenly distributes the material into two streams. Each stream feeds one cone crusher feed bin in parallel. The feed bin discharge goes to vibrating feeder then into the cone crusher. The cone crushers closed side setting (CSS) are 32 mm, with an expected product P_{80} of 32 mm. The cone crusher product is fed to a screen in which the oversize is recycled to the cone diverter chute whilst the undersize is conveyed to a crushed ore stockpile.

The stockpiled ore is conveyed from the crushing and ore sorting plant to a crushed ore stockpile which is then transported by a long distance conveyor to the processing plant.

The stockpiled ore is reclaimed by apron feeders and conveyed to the grinding circuit where it is evenly distributed into two streams by a diverter chute. The grinding circuit consists of two high pressure grinding rolls (HGPR) in parallel. The HPGRs are fed via HGPR apron feeders. The HPGR target crush size is 5 mm. The product is in closed circuit with a double deck banana screen and produces three size fractions. The oversize material is recycled back to the diverter chute, the middle size fraction stream reports to dense media separation and the undersize fraction stream reports to agglomeration. Historical testwork suggests the copper may concentrate in the finer fractions, so a provisional bypass stream has been included that will see the undersize fraction from the double deck screen report directly to the ball mill for subsequent flotation. HPGR introduces micro-cracking that improves leach kinetics, allowing for maximum metal extraction during the heap leach process.

Ore Sorting

The location of an ore sorter is dependent on the ore sorting technology selected. Two ore sorting technologies have been identified for use on chalcopyrite deposits: magnetic resonance (MR) and x-ray transmission (XRT) sorting. MR ore sorters use mechanical diverters to separate valuable minerals from gangue, and are typically performed on coarse particles (~200 mm). This would limit the MR ore sorter to the primary crusher product conveyor. If an MR ore sorter were to be used, then the open side setting on the gyratory crusher would need to be re-evaluated.

XRT ore sorters typically use pneumatic diverters via air jetting to separate gangue. This requires a specific size range to ensure the air jets can effectively separate the particles. Secondary crushed product would be required to feed an XRT ore sorter. If an XRT sorter were selected, the secondary cone crusher CSS may need to be increased.

17.2.2 Single Stage DMS and Grinding

Dense medium separation (DMS) is a gravity separation method that separates products based on the specific gravity (SG) of a particle. This is done by selectively elevating the medium density in the cyclone to a desired SG by the addition of ferrosilicon (FeSi). The particles above the selected SG will sink whilst the particles below will float. The middle size fraction from the HGPR grinding circuit reports to DMS. The material is passed through a series of dense medium separation cyclones with the medium at a SG of 2.75. Particles with a composite density greater than the medium SG will sink (‘sinks’) and composite particles with a density less than the medium SG will float (‘floats’). The sinks are sent to a ball mill for grinding prior to flotation and the floats are sent to the agglomeration drum prior to heap leaching.

The ball mill will be an overflow type in a closed circuit with an installed motor of 1,400 kW. The mill will be fed at F_{80} of 5,000 µm. The trammel undersize is passed into the discharge hopper whilst the scats are sent to the scats bin. The product is fed into a cluster of eight hydrocyclone clusters which will operate with a cut size of 75 µm. The cyclone underflow is recycled back into the ball mill for further processing whilst the cyclone overflow is sent to the flotation circuit.

17.2.3 Flotation

The cyclone overflow is treated with a long chained xanthate collector at a low concentration (50 g/t). Potassium amyl xanthate (PAX) is commonly used and will be used for conditioning of the ore prior to flotation in a rougher cell. PAX is suitable due to the high collection power. PAX will additionally be used for the scavenger cell but it will be added at a higher concentration (100 g/t) to improve the copper recovery. A short chained collector will be used for the cleaner and recleaner cells, possibly sodium ethyl xanthate (SEX), due to its high selectivity. SEX will be added at a concentration of 50 g/t in cleaner flotation and at concentration of (50 g/t) in re-cleaner flotation to improve copper grade.

The ore from the hydrocyclone overflow treated in a rougher conditioning tank. Flotation reagents are mixed and agitated with slurry at 35%wt solids with a residence time of 10 minutes. The product of the conditioning tank proceeds to the rougher flotation cell.

Air is sparged into the rougher flotation cell, with sulphide minerals attaching to the bubbles and floating to the top. The mineralised froth is collected and sent to the cleaner conditioning tank for further processing. The tailings exit the tank and report to the scavenger flotation conditioning tank. Flotation reagents are mixed and agitated with slurry at 35%wt solids with a residence time of 5 minutes. It is then sent to the scavenger flotation cell where the residence time is 30 minutes. The mineralised froth is reported to the cleaner conditioning tank and the tails are sent to the tailings thickener feed tank.

The concentrates from the rougher and scavenger are combined at cleaner conditioning tank with flotation reagents and conditioned to 35%wt solids with a residence time of 5 minutes. This is then reported to the cleaner flotation cell where the residence time is 15 minutes. The mineralised froth is then reported to the re-cleaner conditioning tank and the tailings to the regrind mill. The cleaner concentrate is combined with flotation reagents at 35%wt solids with a residence time of 5 minutes. This is then reported to the re-cleaner flotation column where the residence time is 5 minutes. The mineralised froth is then reported to the concentrate thickener and the tailings to the regrind mill.

The regrind mill is in a closed circuit arrangement with an assumed ball charge of 80% and 1.8 mm ball size. The tails are reported to the rougher conditioning tank after grinding to recycle the now liberated copper particles. The concentrate is left to settle in the concentrate thickener. The concentrate thickener has an underflow solids content of 50%wt and is reported to the concentrate filter feed tank and is sent to the belt filter. The overflow is sent to the concentrate thickener overflow tank (300-TK-05) and sent to the process water dam. The belt filter produces a concentrate cake at 10% moisture which is sent to a radial stacker (300-CV-01) to store in a concentrate pile. The filtrate is sent to a tank and recycled as belt wash water.

Direct Sale

Concentrate can be treated or sold directly. If sold directly, cash flow can be brought forward, which would typically be delayed due to the heap leach. The product will be predominantly chalcopyrite ($CuFeS_2$) with a few minor impurities which will affect the end cost price. Concentrate grade and shipment costs will be the key factors that dictate the feasibility of direct sale.

Treatment Option 1: Leaching and Solvent Extraction

The chalcopyrite concentrate is transferred into a doping tank by a conveyor where copper sulphate is used to dope the product to improve the leachability of the concentrate seen in Figure 17-1. The doped concentrate is transferred into a mixing tank where sulphuric acid is introduced and mixed. The concentrate is transferred to a series of continuous stirred tank reactors (CSTR) with media scrubbers in between to break down the particles and allow for improved leaching. Chalcopyrite tends to passivate and prevents acid from attacking the surface. The media scrubbers between tanks attrition the surface of the particles ensuring the leaching process continues. The final solution is then transferred into the copper solvent extraction circuit for further processing. This treatment option reduces handling and transport costs and increases overall copper recovery.

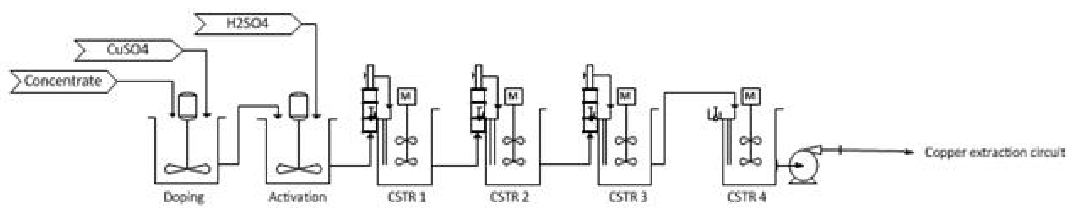


Figure 17-1: Method of concentrate leaching.

Treatment Option 2: Super Concentrate (Chalcocite)

There is a possibility to improve the concentrate by means of converting chalcopyrite (34.6 % Cu) to chalcocite (63.5 % Cu), which has a much higher copper grade. The chalcopyrite concentrate is fed to an autoclave system seen in Figure 17-2, which reduces the impurities (such as iron) and increase the copper content of the concentrate. The process involves reacting chalcopyrite ($CuFeS_2$) with brine and copper sulphate in a NONOX autoclave (non-oxidative, high temperature environment). The ‘super concentrate’ product is sent to a filter and the filtrate is sent to a treatment plant where limestone is used to clean the solution and recycle back into the NONOX. This process would increase the sale value of the concentrate and also reduce the shipping tonnages.

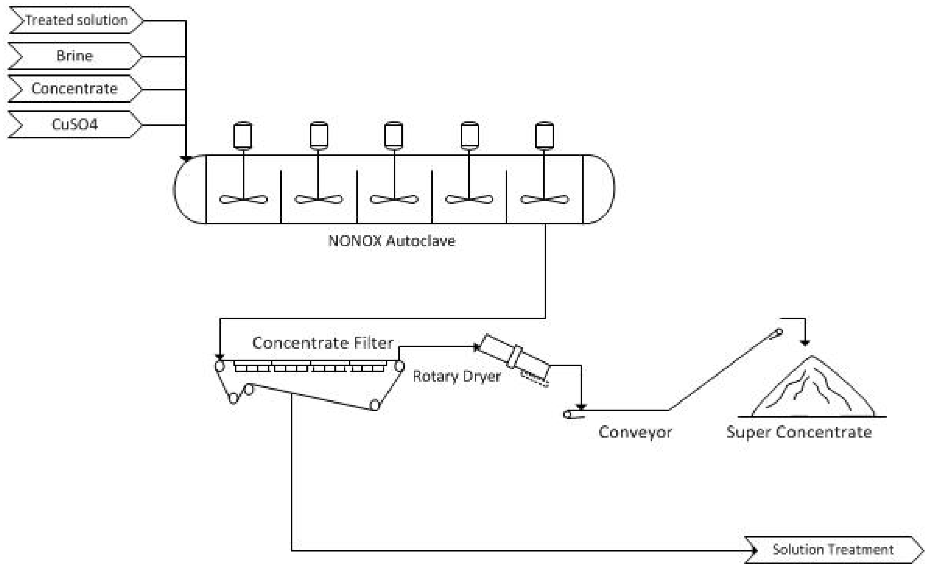


Figure 17-2: Production of 'super concentrate'.

For economic analysis it was assumed that direct sale of the concentrate would be the selected method. Further investigation into concentrate upgrade methods would be required.

17.2.4 Agglomeration Drum

Agglomeration improves the permeability of the heap and facilitates even acid flow without pooling and increasing the amount of oxygen available for reaction. Additionally, pre-wetting will reduce the losses of fines from the wind and increase the leaching kinetics of the ore. Heap leaching requires good percolation throughout the heap to ensure maximum metal recovery is realised. Clays and fine particles can hinder solution flow through the heap, and the ore is often agglomerated to overcome this issue.

The undersize particles from HPGR and the DMS floats are combined with binder, sulphuric acid and water to agglomerate the ore into clumps. It is considered essential to undergo agglomeration. The binder and is added to the agglomeration drum in powder form. Additionally, it is a requirement to add 98% sulphuric acid to pre-leach the chalcopyrite and reduce heap leach times.

17.2.5 Heap Leach

The ore will be stacked by inclined conveyor stackers, producing a heap. This is a preferred stacking method due to conveyor stacking being able to reduce ore segregation which allows for increased permeability. Due to the use of sulphuric acid the conveyor edges must be moulded, open edge belts will severely corrode. Additionally, it is preferable to splice the conveyor belt instead of using clips as it reduces spillage and belt stress.

Drip lines are used primarily in arid environments due to the substantially reduced evaporation in comparison to heap sprays. The drip lines are buried

10 cm to 50 cm beneath the surface of the heap to minimise evaporation. The irrigation rate will be approximately 4-8 $L/h/m^2$. The primary heap pad will be irrigated with solution from the intermediate leach solution (ILS) pond. The secondary and the wash heap pad will be irrigated with solution from the barren pond.

The pad will require a double liner (HDPE) to minimise any possible loss of liquid from liner punctures. Due to the high evaporation rate in the area and close proximity of a river, there is a strict need to minimise risk of the heap solution entering the environment.

Primary Heap

The primary heap will consist of fresh ore from the agglomeration drum that is stacked using conveyors and irrigated from the intermediate leaching solution (ILS) pond. The ILS pond will contain a low concentration leached solution from the secondary pad. The primary heap is leached for 120 days and the pregnant leach solution (PLS) from the primary heap is collected in the pregnant solution pond. The leached ore then becomes the secondary heap by rerouting the flow of the particular piping.

Secondary Heap

The secondary heap will be irrigated from the barren solution pond. The barren pond solution contains leftover metal sulphates from the solvent extraction raffinate. The ILS from the secondary heap is collected in the ILS pond after the ore is spent. The spent ore becomes the washing heap by rerouting the flow of particular piping.

Washing Heap

The washing heap will be irrigated with solution from the barren pond. This ore is washed with solution through drip irrigation periodically (can be conducted over several years). The solution from the heap is collected in the barren pond and used for leaching of the secondary heap.

17.2.6 Crud Removal Systems

Several operations have installed pinned-bed clarifiers on the PLS streams and have been effective, there are examples where the total suspended solids are consistently reduced to <20 mg/L. This is effective as the uncontrolled separation of solids from the process liquor is usually a significant contributor to crud formation.

17.2.7 Copper Solvent Extraction/Electrowinning

The copper solvent extraction (SX) circuit will consist of two extraction cells and two stripping cells. Two extraction cells are used due to the high concentration of copper in the solution to extract as much copper into the organic phase as possible.

Solvent extraction works by combining an organic extractant with an aqueous acid leaching solution at a favourable pH to transfer metal ions of interest into the organic phase. The copper depleted aqueous phase is referred to as the raffinate is sent to the next circuit. The extraction of copper from dilute sulphuric acid is pH dependent with most copper SX being performed at a pH of 2. Due to the similarities in acid dissociation constants the iron in solution will have to be monitored and subsequently removed to improve the copper grade in the end product.

Extraction

In the extraction stages the PLS solution is mixed with organic solution containing extractant. The extractant releases its protons and coordinates with copper, transferring the copper from an aqueous phase to organic phase as an extractant complex. The protons released increase the acid level.

$$Cu^{2+}(a\ ) + 2R\ \ (o\ \ \ ) \rightarrow R_2C\ \ (o\ \ \ ) + 2H^{+}(a\ )$$

Where,

$Cu^{2+}(a\ )$ - is copper in solution
$R\ \ (o\ \ )$ - is the extractant i.e stripped organic
$R_2C\ \ (o\ \ )$ - is the copper/extractant i.e. loaded organic
$2H^{+}(a\ )$ - is acid in raffinate solution

Stripping

$$R_2C\ \ (o\ \ \ ) + 2H^{+}(a\ ) \rightarrow Cu^{++}(a\ ) + 2R\ \ (o\ \ \ )$$

Stripping may be accomplished by contacting the copper containing (loaded) organic with relatively strong sulphuric acid. In most cases, an excess acid concentration of approximately 50 g/L H_2SO_4 is required to maintain adequate stripping. Spent electrolyte (containing copper) may be used as the stripping agent, and the copper content can be increased to any desired level up to about 100 g/L Cu for use as a strong electrolyte. Stripping of copper occurs only when strongly acidic solution is mixed with the organic copper complex. The complex releases its copper and takes on acid.

Products

The copper sulphate solution can be converted to copper metal via electrowinning. The copper electrolysis process involves electroplating of copper from copper sulphate onto a cathode. This is done by passing a current from an inert anode through the solution which causes the copper to plate out. The metallic copper will then be washed and palletised.

The copper sulphate solution can alternately be sent to an evaporative crystalliser where the water is drawn off to leave behind a saturated copper sulphate solution with blue crystals evolving; copper sulphate crystallises as a pentahydrate ($CuSO_4 \cdot 5H_2O$). This is continuously done and refluxed to obtain a high level of saturation which is sent to a centrifuge to collect the copper sulphate solids product. The solution is recycled back into the stripping cell to recycle, and subsequently retain the uncrystallised copper sulphate. The solid product is sent to a flash dryer where water is further drawn off and the product is then collected into the product bin.

17.2.8 Molybdenum Solvent Extraction

The molybdenum SX stages will be connected in counter-current configuration where the copper raffinate will be introduced into the first extraction mixer-settler and the stripped organic phase will be introduced into the second extraction mixer-settler. An O/A ratio of 1 has been incorporated. The molybdenum raffinate is sent to the iron removal circuit and the molybdenum loaded organic phase is collected into the loaded organic tank. The loaded organic phase is then sent to the stripping mixer-settler where ammonia will be introduced to remove the molybdenum as an ammonium dimolybdate (ADM). The stripped organic phase will be sent to the organic control tank and then recycled back to the extraction circuit whilst the loaded aqueous phase is sent to the evaporative crystalliser. The water is drawn off to leave behind a saturated molybdenum solution ($(NH_4)_2Mo_2O_7$)) which subsequently crystallised and sent to a centrifuge to separate the solid crystals. The solid product is collected and dried. The high temperature drying (~300°C) decomposes the ADM to form molybdenum trioxide (MoO_3) and ammonia gas. The molybdenum product collected in the product bin and the ammonia gas is sent to the ammonia regeneration and recovery.

17.2.9 Iron and Aluminium Precipitation

Iron and aluminium in the ore is approximately 1.8% and 7% respectively which will build up as the process continues. The iron and aluminium build up in the solution needs to be treated before recycling the SX raffinate for heap leaching. The process involves pumping the solution from the copper raffinate return line into the iron precipitation tank where lime is added to adjust the pH. Iron will be present primarily as iron sulphate ($FeSO_4$) which when reacted with lime will produce iron hydroxide ($Fe(OH)_2$). Additionally, aluminium will also be present as a sulphide ($Al_2(SO_4)_3$) and will produce an oxide when precipitated. At an elevated pH the hydroxide will precipitate out of solution as a red insoluble oxide. This will be transferred to the iron tailings thickener where the oxide is collected, filtered and disposed of by dry stacking. The thickener overflow and filtrate will be pumped into the barren solution pond where it can be recycled to the heap leach pad.

17.2.10 Tailings

Dry stacking will be conducted to maximise the amount of water recovered. The tails from the flotation circuit are sent to the tailings thickener. The thickener overflow is sent to the process water dam. The tailings thickener has an underflow solids content of 50%wt and is reported to the tailings filter feed tank and is sent to the pressure filter. The pressure filter produces a tailings cake at 10% moisture which is combined with the solids from iron removal. These solids are disposed of by dry stacking via radial stacker. The filtrate is combined with the thickener overflow and sent to the process water dam.

17.2.11 Water Distribution

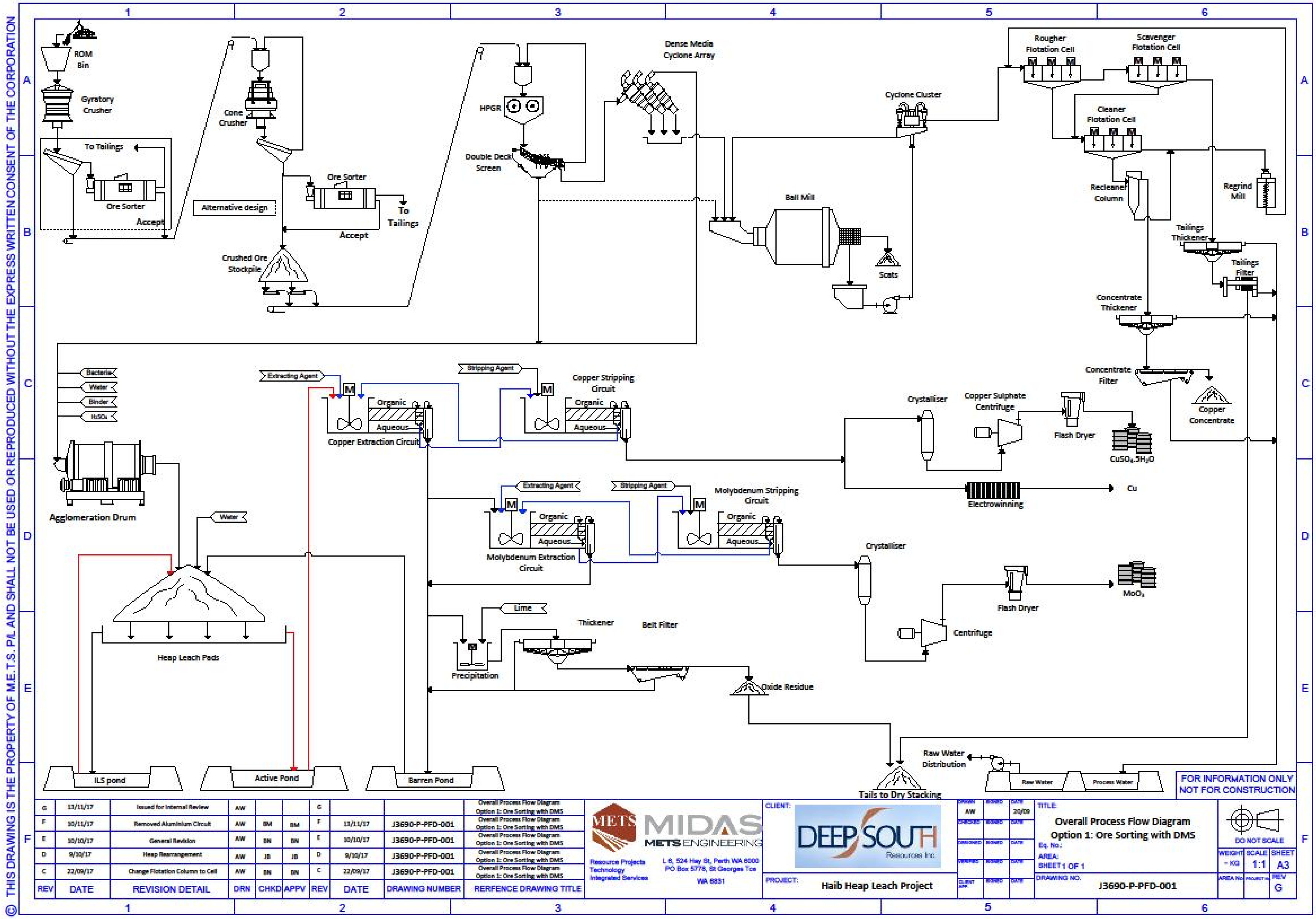
Water distribution covers the raw and process water dams. These will supply general plant water as well as a feed for potable water, fire water, gland seal water, reagents and cooling water.

17.2.12 Reagents

Reagents are mixed in an open area in covered tanks to prevent rain from damaging or reacting with the dry chemicals. The design incorporates accepted methods for mixing, holding, solution distribution and ventilation for each chemical according to MSDS and industry practice. Reagents are kept in a warehouse until they are required. Containment bunds and sump pumps are required for individual reagent handling areas. The sump pumps feed any spilled reagents into the respective tank depending on reagent area. The reagents area will provide storage and distribution for grinding balls, quicklime, flocculant, frother, flotation collectors, frother, sulfuric acid, solvent extraction reagents, binder and FeSi. The current design has an input of elemental sulphur which is burned to produce high concentration sulphuric acid.

17.2.13 Services

A services area will include air distribution (both instrumentation and process air), potable water production using a reverse osmosis package and heavy fuel oil distribution.



ROM Bin
Gyratory Crusher
To Tailings
Ore Sorter
Accept
Alternative design
Cone Crusher
Ore Sorter
To Tailings
Accept
Crushed Ore Stockpile
HPGR
Double Deck Screen
Dense Media Cyclone Array
Cyclone Cluster
Ball Mill
Scats
Rougher Flotation Cell
Scavenger Flotation Cell
Cleaner Flotation Cell
Recleaner Column
Regrind Mill
Tailings Thickener
Tailings Filter
Concentrate Thickener
Concentrate Filter
Copper Concentrate
Bacteria
Water
Binder
H_2SO_4
Extracting Agent
Stripping Agent
Copper Stripping Circuit
Organic
Aqueous
Copper Extraction Circuit
Crystalliser
Copper Sulphate Centrifuge
Flash Dryer
$CuSO_4.5H_2O$
Electrowinning
Cu
Agglomeration Drum
Water
Extracting Agent
Stripping Agent
Molybdenum Stripping Circuit
Molybdenum Extraction Circuit
Crystalliser
Flash Dryer
MoO_3
Centrifuge
Heap Leach Pads
Lime
Thickener
Belt Filter
Precipitation
Oxide Residue
Raw Water Distribution
Tails to Dry Stacking
Raw Water
Process Water
ILS pond
Active Pond
Barren Pond
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REV | DATE | REVISION DETAIL | DRN | CHKD | APPV | REV | DATE | DRAWING NUMBER | REFERENCE DRAWING TITLE
G | 13/11/17 | Issued for Internal Review | AW
F | 10/11/17 | Removed Aluminium Circuit | AW | BM | BM | F | 13/11/17 | J3690-P-PFD-001 | Overall Process Flow Diagram Option 1: Ore Sorting with DMS
E | 10/10/17 | General Revision | AW | BN | BN | E | 10/10/17 | J3690-P-PFD-001 | Overall Process Flow Diagram Option 1: Ore Sorting with DMS
D | 9/10/17 | Heap Rearrangement | AW | JB | JB | D | 9/10/17 | J3690-P-PFD-001 | Overall Process Flow Diagram Option 1: Ore Sorting with DMS
C | 22/09/17 | Change Flotation Column to Cell | AW | BN | BN | C | 22/09/17 | J3690-P-PFD-001 | Overall Process Flow Diagram Option 1: Ore Sorting with DMS
METS MIDAS METS ENGINEERING
Resource Projects Technology Integrated Services
L 6, 524 Hay St, Perth WA 6000 PO Box 5778, St Georges Tce WA 6831
CLIENT:
DEEP SOUTH Resources Inc.
PROJECT: Haib Heap Leach Project
TITLE: Overall Process Flow Diagram Option 1: Ore Sorting with DMS
DO NOT SCALE
SCALE 1:1
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SHEET 1 OF 1
DRAWING NO. J3690-P-PFD-001
REV G

17.3 Process Description – Option 2

17.3.1 Crushing and Ore Handling

Run of Mine (ROM) ore is transported by truck from the mine to the ROM stockpile area near the crushing plant. The material is transferred to a ROM bin, which feeds to a primary crusher. The primary crusher is a gyratory crusher suited to higher crushing capacities. The open side setting (OSS) is expected to be set at 127 mm with an assumed P_{80} of 127 mm to be produced. The output of the gyratory crusher is fed to a diverter chute via a primary conveyor. The diverter chute distributes the material into two streams that feed two cone crusher feed bin in parallel. The feed bin discharge goes to a vibrating feeder then into the cone crushers. The cone crushers closed side setting (CSS) are 32 mm, with an expected product P_{80} of 32 mm. The cone crusher product is fed to a screen in which the oversize is recycled to the diverter chute whilst the undersize is conveyed to a crushed ore stockpile.

The stockpiled crushed ore is conveyed from the crushing plant to the processing plant by a long distance conveyor.

The stockpiled ore reclaimed via apron feeders and transferred by conveyor to the grinding circuit where it is evenly distributed into two streams by a diverter chute. The grinding circuit consists of two high pressure grinding rolls (HGPR) in parallel. The HPGRs are fed via apron feeders. The HPGR target crush size is 5 mm. The product is in closed circuit with a double deck banana screen and produces three size fractions. The oversize material is recycled back to a diverter chute, the middle size fraction stream reports to dense media separation and the undersize fraction stream reports to agglomeration. Historical testwork suggests the copper may concentrate in the finer fractions, so a provisional bypass stream has been included that will see the undersize fraction from the double deck screen reporting directly to the ball mill for subsequent flotation. HPGR introduces micro-cracking that improves leach kinetics, allowing for maximum metal extraction during the heap leach process.

17.3.2 Two-Stage DMS and Grinding

Dense medium separation (DMS) is a gravity separation method that separates products based on the specific gravity (SG) of a particle. This is done by selectively elevating the medium density in the cyclone to a desired SG by the addition of ferrosilicon (FeSi). The particles above the selected SG will sink whilst the particles below will float. The middle size fraction from the cone crushing circuit reports to DMS. Firstly, the material is passed through a series of dense medium separation cyclones with the medium at a SG of 2.65. Particles with a composite density greater than the medium SG will sink (‘sinks’) and composite particles with a density less than the medium SG will float (‘floats’). The floats are sent to tailings and the sinks report to the second dense medium separation cyclone at a medium of 2.75 SG. The sinks are sent to a ball mill for grinding prior to flotation and the floats are sent to the agglomeration drum prior to heap leaching.

The ball mill will be an overflow type in a closed circuit with an installed motor of 1,400 kW. The mill will be fed at F_{80} of 5,000 µm and is assumed to produce a P_{80} of 75 µm. The undersize is passed into the discharge hopper whilst the scats are sent to the scats bin. The product is fed into a cluster of eight hydrocyclones cluster which will operate with a cut size of 75 µm. The cyclone underflow is recycled back into the ball mill for further processing whilst the cyclone overflow is sent to the flotation circuit.

17.3.3 Flotation

The cyclone overflow is treated with a long chained xanthate collector at a low concentration (50 g/t). Potassium amyl xanthate (PAX) is commonly used and will be used for conditioning of the ore prior to flotation in a rougher cell. PAX is suitable due to the high collection power. PAX will additionally be used for the scavenger cell but it will be added at a higher concentration (100 g/t) to improve the copper recovery. A short chained collector will be used for the cleaner and recleaner cells, possibly sodium ethyl xanthate (SEX), due to its high selectivity. SEX will be added at a concentration of 50 g/t in cleaner flotation and at concentration of (50 g/t) in re-cleaner flotation to improve copper grade.

The ore from the hydrocyclone overflow treated in a rougher conditioning tank. Flotation reagents are mixed and agitated with slurry at 35%wt solids with a residence time of 10 minutes. The product of the conditioning tank proceeds to the rougher flotation cell.

Air is sparged into the rougher flotation cell, with sulphide minerals attaching to the bubbles and floating to the top. The mineralised froth is collected and sent to the cleaner conditioning tank for further processing. The tailings exit the tank and report to the scavenger flotation conditioning tank. Flotation reagents are mixed and agitated with slurry at 35%wt solids with a residence time of 5 minutes. It is then sent to the scavenger flotation cell where the residence time is 30 minutes. The mineralised froth is reported to the cleaner conditioning tank and the tails are sent to the tailings thickener feed tank.

The concentrates from the rougher and scavenger are combined at cleaner conditioning tank with flotation reagents and conditioned to 35%wt solids with a residence time of 5 minutes. This is then reported to the cleaner flotation cell where the residence time is 15 minutes. The mineralised froth is then reported to the re-cleaner conditioning tank and the tailings to the regrind mill. The cleaner concentrate is combined with flotation reagents at 35%wt solids with a residence time of 5 minutes. This is then reported to the re-cleaner flotation column where the residence time is 5 minutes. The mineralised froth is then reported to the concentrate thickener and the tailings to the regrind mill.

The regrind mill is in a closed circuit arrangement with an assumed ball charge of 80% and 1.8 mm ball size. The tails are reported to the rougher conditioning tank after grinding to recycle the now liberated copper particles. The concentrate is left to settle in the concentrate thickener. The concentrate thickener has an underflow solids content of 50%wt and is reported to the concentrate filter feed tank and is sent to the belt filter. The overflow is sent to the concentrate thickener overflow tank (300-TK-05) and sent to the process water dam. The belt filter produces a concentrate cake at 10% moisture which is sent to a radial stacker (300-CV-01) to store in a concentrate pile. The filtrate is sent to a tank and recycled as belt wash water.

Direct Sale

Concentrate can be treated or sold directly. If sold directly, cash flow can be brought forward, which would typically be delayed due to the heap leach. The product will be predominantly chalcopyrite ($CuFeS_2$) with a few minor impurities which will affect the end cost price. Concentrate grade and shipment costs will be the key factors that dictate the feasibility of direct sale.

Treatment Option 1: Leaching and Solvent Extraction

The chalcopyrite concentrate is transferred into a doping tank by a conveyor where copper sulphate is used to dope the product to improve the leachability of the concentrate seen in Figure 17-3. The doped concentrate is transferred into a mixing tank where sulphuric acid is introduced and mixed. The concentrate is transferred to a series of continuous stirred tank reactors (CSTR) with media scrubbers in between to break down the particles and allow for improved leaching. Chalcopyrite tends to passivate and prevents acid from attacking the surface. The media scrubbers between tanks attrition the surface of the particles ensuring the leaching process continues. The final solution is then transferred into the copper solvent extraction circuit for further processing. This treatment option reduces handling and transport costs and increases overall copper recovery.

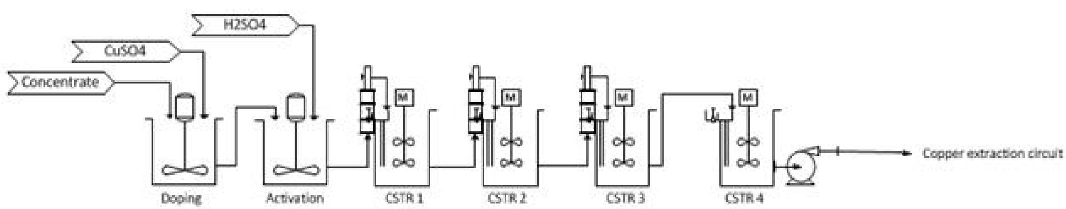


Figure 17-3: Method of concentrate leaching.

Treatment Option 2: Super Concentrate (Chalcocite)

There is a possibility to improve the concentrate by means of converting chalcopyrite (34.6 % Cu) to chalcocite (63.5 % Cu), which has a much higher copper grade. The chalcopyrite concentrate is fed to an autoclave system seen in Figure 17-4, which reduces the impurities (such as iron) and increase the copper content of the concentrate. The process involves reacting chalcopyrite ($CuFeS_2$) with brine and copper sulphate in a NONOX autoclave (non-oxidative, high temperature environment). The 'super concentrate' product is sent to a filter and the filtrate is sent to a treatment plant where limestone is used to clean the solution and recycle back into the NONOX. This process would increase the sale value of the concentrate and also reduce the shipping tonnages.

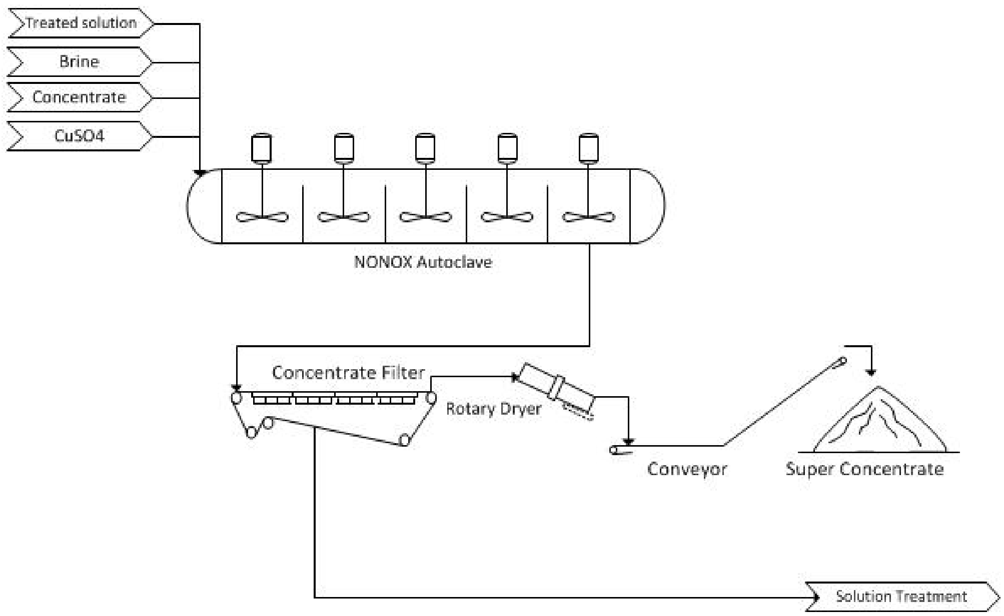


Figure 17-4: Production of 'super concentrate'.

For economic analysis it was assumed that direct sale of the concentrate would be the selected method. Further investigation into concentrate upgrade methods would be required.

17.3.4 Agglomeration Drum

Agglomeration improves the permeability of the heap and facilitates even acid flow without pooling and increasing the amount of oxygen available for reaction. Additionally, pre-wetting will reduce the losses of fines from the wind and increase the leaching kinetics of the ore. Heap leaching requires good percolation throughout the heap to ensure maximum metal recovery is realised. Clays and fine particles can hinder solution flow through the heap, and the ore is often agglomerated to overcome this issue.

The undersize particles from HPGR and the DMS floats are combined with binder, sulphuric acid and water to agglomerate the ore into clumps. It is considered essential to undergo agglomeration. The binder and is added to the agglomeration drum in powder form. Additionally, it is a requirement to add 98% sulphuric acid to pre-leach the chalcopyrite and reduce heap leach times.

17.3.5 Heap Leach

The ore will be stacked by inclined conveyor stackers, producing a heap. This is a preferred stacking method due to conveyor stacking being able to reduce ore segregation which allows for increased permeability. Due to the use of sulphuric acid the conveyor edges must be moulded, open edge belts will severely corrode. Additionally, it is preferable to splice the conveyor belt instead of using clips as it reduces spillage and belt stress.

Drip lines are used primarily in arid environments due to the substantially reduced evaporation in comparison to heap sprays. The drip lines are buried 10 cm to 50 cm beneath the surface of the heap to minimise evaporation. The irrigation rate will be approximately 4-8 L/h/m^2. The primary heap pad will be irrigated with solution from the intermediate leach solution (ILS) pond. The secondary and the wash heap pad will be irrigated with solution from the barren pond.

The pad will require a double liner (HDPE) to minimise any possible loss of liquid from liner punctures. Due to the high evaporation rate in the area and close proximity of a river, there is a strict need to minimise risk of the heap solution entering the environment.

Primary Heap

The primary heap will consist of fresh ore from the agglomeration drum that is stacked using conveyors and irrigated from the intermediate leaching solution (ILS) pond. The ILS pond will contain a low concentration leached solution from the secondary pad. The primary heap is leached for 120 days and the pregnant leach solution (PLS) from the primary heap is collected in the pregnant solution pond. The leached ore then becomes the secondary heap by rerouting the flow of the particular piping.

Secondary Heap

The secondary heap will be irrigated from the barren solution pond. The barren pond solution contains leftover metal sulphates from the solvent extraction raffinate. The ILS from the secondary heap is collected in the ILS pond after the ore is spent. The spent ore becomes the washing heap by rerouting the flow of particular piping.

Washing Heap

The washing heap will be irrigated with solution from the barren pond. This ore is washed with solution through drip irrigation periodically (can be conducted over several years). The solution from the heap is collected in the barren pond and used for leaching of the secondary heap.

17.3.6 Crud Removal Systems

Several operations have installed pinned-bed clarifiers on the PLS streams and have been effective, there are examples where the total suspended solids are consistently reduced to <20 mg/L. This is effective as the uncontrolled separation of solids from the process liquor is usually a significant contributor to crud formation.

17.3.7 Copper Solvent Extraction/Electrowinning

The copper solvent extraction (SX) circuit will consist of two extraction cells and two stripping cells. Two extraction cells are used due to the high concentration of copper in the solution to extract as much copper into the organic phase as possible.

Solvent extraction works by combining an organic extractant with an aqueous acid leaching solution at a favourable pH to transfer metal ions of interest into the organic phase. The copper depleted aqueous phase is referred to as the raffinate is sent to the next circuit. The extraction of copper from dilute sulphuric acid is pH dependent with most copper SX being performed at a pH of 2. Due to the similarities in acid dissociation constants the iron in solution will have to be monitored and subsequently removed to improve the copper grade in the end product.

Extraction

In the extraction stages the PLS solution is mixed with organic solution containing extractant. The extractant releases its protons and coordinates with copper, transferring the copper from an aqueous phase to organic phase as an extractant complex. The protons released increase the acid level.

$$Cu^{2+}(a\ ) + 2R\ \ (o\ \ ) \rightarrow R_2C\ \ (o\ \ ) + 2H^{+}(a\ )$$

Where,
$Cu^{2+}(a\ )$ - is copper in solution
$R\ \ (o\ \ )$ - is the extractant i.e stripped organic
$R_2C\ \ (o\ \ )$ - is the copper/extractant i.e. loaded organic
$2H^{+}(a\ )$ - is acid in raffinate solution

Stripping

$$R_2C\ \ (o\ \ ) + 2H^{+}(a\ ) \rightarrow Cu^{++}(a\ ) + 2R\ \ (o\ \ )$$

Stripping may be accomplished by contacting the copper containing (loaded) organic with relatively strong sulphuric acid. In most cases, an excess acid concentration of approximately 50 g/L H_2SO_4 is required to maintain adequate stripping. Spent electrolyte (containing copper) may be used as the stripping agent, and the copper content can be increased to any desired level up to about 100 g/L Cu for use as a strong electrolyte. Stripping of copper occurs only when strongly acidic solution is mixed with the organic copper complex. The complex releases its copper and takes on acid.

Products

The copper sulphate solution can be converted to copper metal via electrowinning. The copper electrolysis process involves electroplating of copper from copper sulphate onto a cathode. This is done by passing a current from an inert anode through the solution which causes the copper to plate out. The metallic copper will then be washed and palletised.

The copper sulphate solution can alternately be sent to an evaporative crystalliser where the water is drawn off to leave behind a saturated copper sulphate solution with blue crystals evolving; copper sulphate crystallises as a pentahydrate ($CuSO_4 \bullet 5H_2O$). This is continuously done and refluxed to obtain a high level of saturation which is sent to a centrifuge to collect the copper sulphate solids product. The solution is recycled back into the stripping cell to recycle, and subsequently retain the uncrystallised copper sulphate. The solid product is sent to a flash dryer where water is further drawn off and the product is then collected into the product bin.

17.3.8 Molybdenum Solvent Extraction

The molybdenum SX stages will be connected in counter-current configuration where the copper raffinate will be introduced into the first extraction mixer-settler and the stripped organic phase will be introduced into the second extraction mixer-settler. An O/A ratio of 1 has been incorporated. The molybdenum raffinate is sent to the iron removal circuit and the molybdenum loaded organic phase is collected into the loaded organic tank. The loaded organic phase is then sent to the stripping mixer-settler where ammonia will be introduced to remove the molybdenum as an ammonium dimolybdate (ADM). The stripped organic phase will be sent to the organic control tank and then recycled back to the extraction circuit whilst the loaded aqueous phase is sent to the evaporative crystalliser. The water is drawn off to leave behind a saturated molybdenum solution ($(NH_4)_2Mo_2O_7$)) which subsequently crystallised and sent to a centrifuge to separate the solid crystals. The solid product is collected and dried. The high temperature drying (~300°C) decomposes the ADM to form molybdenum trioxide (MoO_3) and ammonia gas. The molybdenum product collected in the product bin and the ammonia gas is sent to the ammonia regeneration and recovery.

17.3.9 Iron and Aluminium Precipitation

Iron and aluminium in the ore is approximately 1.8% and 7% respectively which will build up as the process continues. The iron and aluminium build up in the solution needs to be treated before recycling the SX raffinate for heap leaching. The process involves pumping the solution from the copper raffinate return line into the iron precipitation tank where lime is added to adjust the pH. Iron will be present primarily as iron sulphate ($FeSO_4$) which when reacted with lime will produce iron hydroxide ($Fe(OH)_2$). Additionally, aluminium will also be present as a sulphide ($Al_2(SO_4)_3$) and will produce an oxide when precipitated. At an elevated pH the hydroxide will precipitate out of solution as a red insoluble oxide. This will be transferred to the iron tailings thickener where the oxide is collected, filtered and disposed of by dry stacking. The thickener overflow and filtrate will be pumped into the barren solution pond where it can be recycled to the heap leach pad.

17.3.10 Tailings

Dry stacking will be conducted to maximise the amount of water recovered. The tails from the flotation circuit are sent to the tailings thickener. The thickener overflow is sent to the process water dam. The tailings thickener has an underflow solids content of 50%wt and is reported to the tailings filter feed tank and is sent to the pressure filter. The pressure filter produces a tailings cake at 10% moisture which is combined with the solids from iron removal. These solids are disposed of by dry stacking via radial stacker. The filtrate is combined with the thickener overflow and sent to the process water dam.

17.3.11 Water Distribution

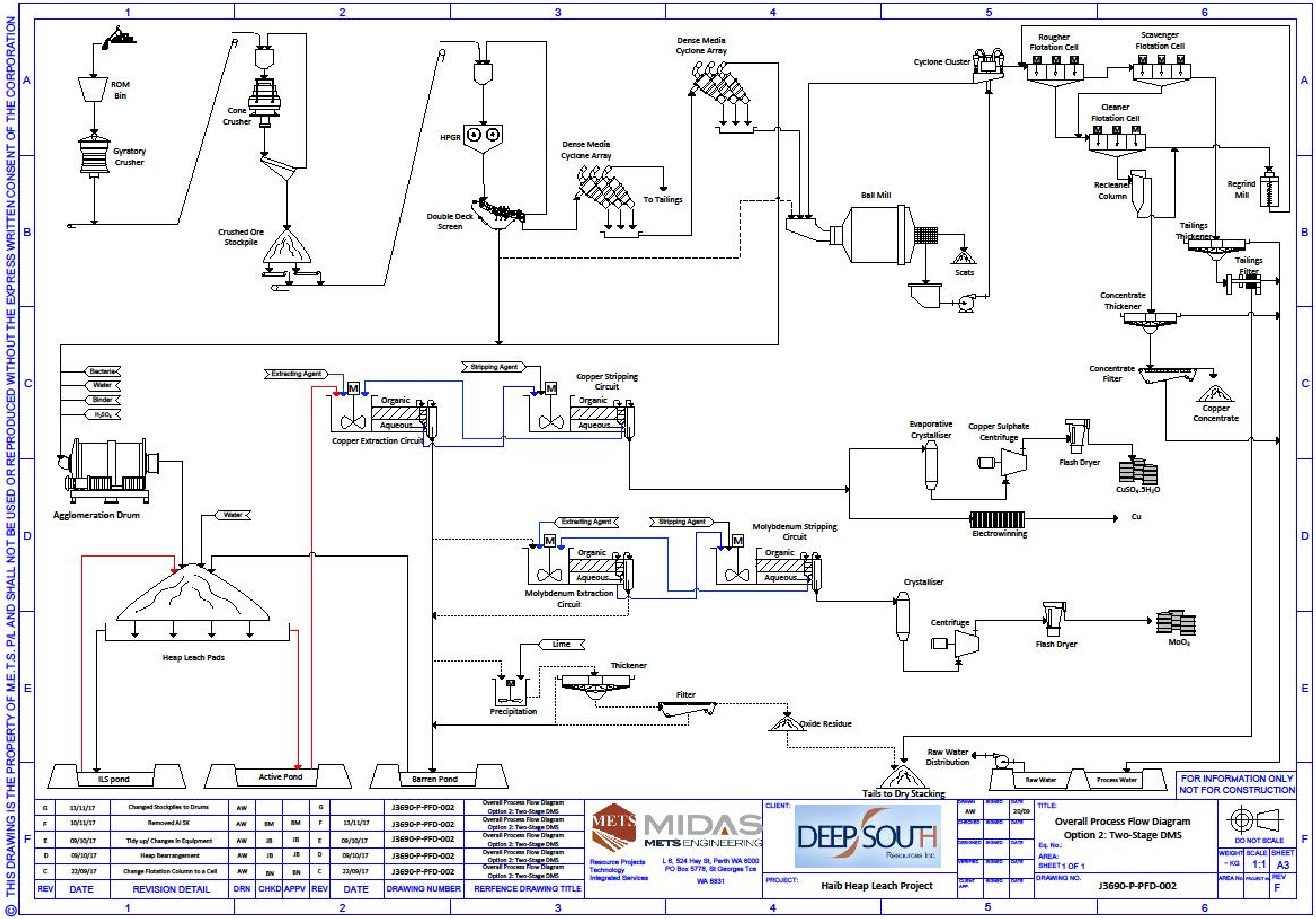
Water distribution covers the raw and process water dams. These will supply general plant water as well as a feed for potable water, fire water, gland seal water, reagents and cooling water.

17.3.12 Reagents

Reagents are mixed in an open area in covered tanks to prevent rain from damaging or reacting with the dry chemicals. The design incorporates accepted methods for mixing, holding, solution distribution and ventilation for each chemical according to MSDS and industry practice. Reagents are kept in a warehouse until they are required. Containment bunds and sump pumps are required for individual reagent handling areas. The sump pumps feed any spilled reagents into the respective tank depending on reagent area. The reagents area will provide storage and distribution for grinding balls, quicklime, flocculant, frother, flotation collectors, frother, sulfuric acid, solvent extraction reagents, binder and FeSi. The current design has an input of elemental sulphur which is burned to produce high concentration sulphuric acid.

17.3.13 Services

A services area will include air distribution (both instrumentation and process air), potable water production using a reverse osmosis package and heavy fuel oil distribution.



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Tailings Filter
Concentrate Thickener
Concentrate Filter
Copper Concentrate
Bacteria
Water
Binder
H_2SO_4
Agglomeration Drum
Extracting Agent
Organic
Aqueous
Copper Extraction Circuit
Stripping Agent
Copper Stripping Circuit
Evaporative Crystalliser
Copper Sulphate Centrifuge
Flash Dryer
$CuSO_4.5H_2O$
Electrowinning
Cu
Water
Heap Leach Pads
Molybdenum Extraction Circuit
Molybdenum Stripping Circuit
Crystalliser
Centrifuge
Flash Dryer
MoO_3
Lime
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G 13/11/17 Changed Stockpiles to Drums AW G J3690-P-PFD-002 Overall Process Flow Diagram Option 2: Two-Stage DMS
F 10/11/17 Removed AI SX AW BM BM F 13/11/17 J3690-P-PFD-002 Overall Process Flow Diagram Option 2: Two-Stage DMS
E 09/10/17 Tidy up/ Changes in Equipment AW JB JB E 09/10/17 J3690-P-PFD-002 Overall Process Flow Diagram Option 2: Two-Stage DMS
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METS ENGINEERING
Resource Projects Technology Integrated Services
L 6, 524 Hay St, Perth WA 6000 PO Box 5778, St Georges Tce WA 6831
CLIENT:
DEEP SOUTH Resources Inc.
PROJECT: Haib Heap Leach Project
TITLE: Overall Process Flow Diagram Option 2: Two-Stage DMS
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SHEET 1 OF 1
DRAWING NO. J3690-P-PFD-002
REV F
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17.4 Process Description – Option 3

17.4.1 Crushing and Ore Handling

Run of Mine (ROM) ore is transported by truck from the mine to the ROM stockpile area near the crushing plant. The material is transferred to a ROM bin, which feeds to a primary crusher. The primary crusher is a gyratory crusher suited to higher crushing capacities. The open side setting (OSS) is expected to be set at 127 mm with an assumed P_{80} of 127 mm to be produced. The output of the gyratory crusher is fed to a primary crusher screen with an aperture of 20mm. The undersize material goes to secondary crushing circuit via conveyor. The oversize material feeds a group of diverter chutes and conveyors that evenly distribute the material into four streams. Each stream feeds one ore sorter of the ore sorting circuit which consists of four ore sorters arranged in parallel. The material rejected by the ore sorters feeds the reject stockpile via conveyor.

The ore sorter products go to a conveyor and then to a diverter chute that evenly distributes the material into two streams. Each stream feeds one cone crusher feed bin in parallel. The feed bin discharge goes to a cone crusher vibrating feeder then into the cone crushers. The cone crushers closed side setting (CSS) are 32 mm, with an expected product P_{80} of 32 mm. The cone crusher product is fed to a screen in which the oversize is recycled to the diverter chute whilst the undersize is conveyed to a crushed ore stockpile.

The stockpiled crushed ore is conveyed from the crushing and ore sorting plant to the processing plant by a long distance conveyor.

The stockpiled ore is reclaimed by apron feeders and conveyed to the grinding circuit where it is evenly distributed into two streams by a diverter chute. The grinding circuit consists of two high pressure grinding rolls (HGPR) in parallel. Each stream feeds one HGPR feed bin. The HPGRs are fed via HGPR apron feeders. The HPGR target crush size is 5 mm. The product is in closed circuit with a screen. The oversize material is recycled back to the diverter chute and the undersize fraction stream reports to agglomeration. HPGR introduces micro-cracking that improves leach kinetics, allowing for maximum metal extraction during the heap leach process.

17.4.2 Ore Sorting

The location of an ore sorter is dependent on the ore sorting technology selected. Two ore sorting technologies have been identified for use on chalcopyrite deposits: magnetic resonance (MR) and x-ray transmission (XRT) sorting. MR ore sorters use mechanical diverters to separate valuable minerals from gangue, and are typically performed on particles greater than 200 mm. This would limit the MR ore sorter to the primary crusher product conveyor. If an MR ore sorter were to be used, then the open side setting on the gyratory crusher would need to be increased.

XRT ore sorters typically use pneumatic diverters via air jetting to separate gangue. This requires a specific size range to ensure the air jets can effectively separate the particles. Secondary crushed product would be required to feed an XRT ore sorter. If an XRT sorter were selected, the secondary cone crusher CSS would need to be increased.

17.4.3 Agglomeration Drum

Agglomeration improves the permeability of the heap and facilitates even acid flow without pooling and increasing the amount of oxygen available for reaction. Additionally, pre-wetting will reduce the losses of fines from the wind and increase the leaching kinetics of the ore. Heap leaching requires good percolation throughout the heap to ensure maximum metal recovery is realised. Clays and fine particles can hinder solution flow through the heap, and the ore is often agglomerated to overcome this issue.

The undersize particles from HPGR and the DMS floats are combined with binder, sulphuric acid and water to agglomerate the ore into clumps. It is considered essential to undergo agglomeration. The binder and is added to the agglomeration drum in powder form. Additionally, it is a requirement to add 98% sulphuric acid to pre-leach the chalcopyrite and reduce heap leach times.

17.4.4 Heap Leach

The ore will be stacked by inclined conveyor stackers, producing a heap. This is a preferred stacking method due to conveyor stacking being able to reduce ore segregation which allows for increased permeability. Due to the use of sulphuric acid the conveyor edges must be moulded, open edge belts will severely corrode. Additionally, it is preferable to splice the conveyor belt instead of using clips as it reduces spillage and belt stress.

Drip lines are used primarily in arid environments due to the substantially reduced evaporation in comparison to heap sprays. The drip lines are buried 10 cm to 50 cm beneath the surface of the heap to minimise evaporation. The irrigation rate will be approximately 4-8 $L/h/m^2$. The primary heap pad will be irrigated with solution from the intermediate leach solution (ILS) pond. The secondary and the wash heap pad will be irrigated with solution from the barren pond.

The pad will require a double liner (HDPE) to minimise any possible loss of liquid from liner punctures. Due to the high evaporation rate in the area and close proximity of a river, there is a strict need to minimise risk of the heap solution entering the environment.

Primary Heap

The primary heap will consist of fresh ore from the agglomeration drum that is stacked using conveyors and irrigated from the intermediate leaching solution (ILS) pond. The ILS pond will contain a low concentration leached solution from the secondary pad. The primary heap is leached for 120 days and the pregnant leach solution (PLS) from the primary heap is collected in the pregnant solution pond. The leached ore then becomes the secondary heap by rerouting the flow of the particular piping.

Secondary Heap

The secondary heap will be irrigated from the barren solution pond. The barren pond solution contains leftover metal sulphates from the solvent extraction raffinate. The ILS from the secondary heap is collected in the ILS pond after the ore is spent. The spent ore becomes the washing heap by rerouting the flow of particular piping.

Washing Heap

The washing heap will be irrigated with solution from the barren pond. This ore is washed with solution through drip irrigation periodically (can be conducted over several years). The solution from the heap is collected in the barren pond and used for leaching of the secondary heap.

17.4.5 Crud Removal Systems

Several operations have installed pinned-bed clarifiers on the PLS streams and have been effective, there are examples where the total suspended solids are consistently reduced to <20 mg/L. This is effective as the uncontrolled separation of solids from the process liquor is usually a significant contributor to crud formation.

17.4.6 Copper Solvent Extraction/Electrowinning

The copper solvent extraction (SX) circuit will consist of two extraction cells and two stripping cells. Two extraction cells are used due to the high concentration of copper in the solution to extract as much copper into the organic phase as possible.

Solvent extraction works by combining an organic extractant with an aqueous acid leaching solution at a favourable pH to transfer metal ions of interest into the organic phase. The copper depleted aqueous phase is referred to as the raffinate is sent to the next circuit. The extraction of copper from dilute sulphuric acid is pH dependent with most copper SX being performed at a pH of 2. Due to the similarities in acid dissociation constants the iron in solution will have to be monitored and subsequently removed to improve the copper grade in the end product.

Extraction

In the extraction stages the PLS solution is mixed with organic solution containing extractant. The extractant releases its protons and coordinates with copper, transferring the copper from an aqueous phase to organic phase as an extractant complex. The protons released increase the acid level.

$$Cu^{2+}(a\ ) + 2R\ \ (o\ \ ) \rightarrow R_2C\ \ (o\ \ ) + 2H^{+}(a\ )$$

Where,
$Cu^{2+}(a\ )$ - is copper in solution
$R\ \ (o\ \ )$ - is the extractant i.e stripped organic
$R_2C\ \ (o\ \ )$ - is the copper/extractant i.e. loaded organic

$2H^{+}(a\ )$ - is acid in raffinate solution

Stripping

$$R_2C\ (o\ \ ) + 2H^{+}(a\ ) \rightarrow Cu^{++}(a\ ) + 2R\ (o\ \ )$$

Stripping may be accomplished by contacting the copper containing (loaded) organic with relatively strong sulphuric acid. In most cases, an excess acid concentration of approximately 50 g/L H_2SO_4 is required to maintain adequate stripping. Spent electrolyte (containing copper) may be used as the stripping agent, and the copper content can be increased to any desired level up to about 100 g/L Cu for use as a strong electrolyte. Stripping of copper occurs only when strongly acidic solution is mixed with the organic copper complex. The complex releases its copper and takes on acid.

Products

The copper sulphate solution can be converted to copper metal via electrowinning. The copper electrolysis process involves electroplating of copper from copper sulphate onto a cathode. This is done by passing a current from an inert anode through the solution which causes the copper to plate out. The metallic copper will then be washed and palletised.

The copper sulphate solution can alternately be sent to an evaporative crystalliser where the water is drawn off to leave behind a saturated copper sulphate solution with blue crystals evolving; copper sulphate crystallises as a pentahydrate ($CuSO_4 \bullet 5H_2O$). This is continuously done and refluxed to obtain a high level of saturation which is sent to a centrifuge to collect the copper sulphate solids product. The solution is recycled back into the stripping cell to recycle, and subsequently retain the uncrystallised copper sulphate. The solid product is sent to a flash dryer where water is further drawn off and the product is then collected into the product bin.

17.4.7 Molybdenum Solvent Extraction

The molybdenum SX stages will be connected in counter-current configuration where the copper raffinate will be introduced into the first extraction mixer-settler and the stripped organic phase will be introduced into the second extraction mixer-settler. An O/A ratio of 1 has been incorporated. The molybdenum raffinate is sent to the iron removal circuit and the molybdenum loaded organic phase is collected into the loaded organic tank. The loaded organic phase is then sent to the stripping mixer-settler where ammonia will be introduced to remove the molybdenum as an ammonium dimolybdate (ADM). The stripped organic phase will be sent to the organic control tank and then recycled back to the extraction circuit whilst the loaded aqueous phase is sent to the evaporative crystalliser. The water is drawn off to leave behind a saturated molybdenum solution ($(NH_4)_2Mo_2O_7$)) which subsequently crystallised and sent to a centrifuge to separate the solid crystals. The solid product is collected and dried. The high temperature drying (~300°C) decomposes the ADM to form molybdenum trioxide (MoO_3) and ammonia gas. The molybdenum product collected in the product bin and the ammonia gas is sent to the ammonia regeneration and recovery.

17.4.8 Iron and Aluminium Precipitation

Iron and aluminium in the ore is approximately 1.8% and 7% respectively which will build up as the process continues. The iron and aluminium build up in the solution needs to be treated before recycling the SX raffinate for heap leaching. The process involves pumping the solution from the copper raffinate return line into the iron precipitation tank where lime is added to adjust the pH. Iron will be present primarily as iron sulphate ($FeSO_4$) which when reacted with lime will produce iron hydroxide ($Fe(OH)_2$). Additionally, aluminium will also be present as a sulphide ($Al_2(SO_4)_3$) and will produce an oxide when precipitated. At an elevated pH the hydroxide will precipitate out of solution as a red insoluble oxide. This will be transferred to the iron tailings thickener where the oxide is collected, filtered and disposed of by dry stacking. The thickener overflow and filtrate will be pumped into the barren solution pond where it can be recycled to the heap leach pad.

17.4.9 Water Distribution

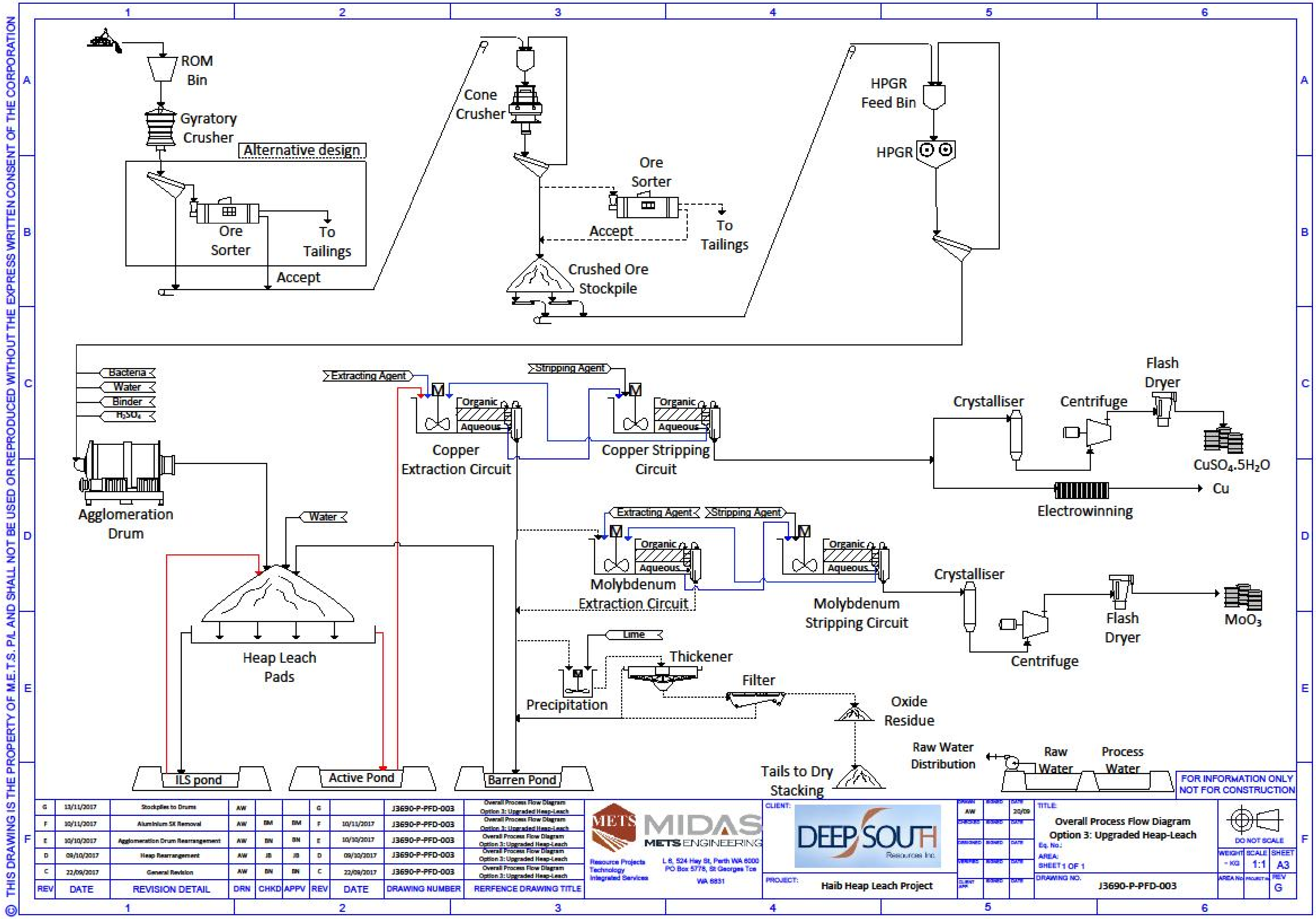
Water distribution covers the raw and process water dams. These will supply general plant water as well as a feed for potable water, fire water, gland seal water, reagents and cooling water.

17.4.10 Reagents

Reagents are mixed in an open area in covered tanks to prevent rain from damaging or reacting with the dry chemicals. The design incorporates accepted methods for mixing, holding, solution distribution and ventilation for each chemical according to MSDS and industry practice. Reagents are kept in a warehouse until they are required. Containment bunds and sump pumps are required for individual reagent handling areas. The sump pumps feed any spilled reagents into the respective tank depending on reagent area. The reagents area will provide storage and distribution for quicklime, flocculant, sulfuric acid, solvent extraction reagents and binder. The current design has an input of elemental sulphur which is burned to produce high concentration sulphuric acid.

17.4.11 Services

A services area will include air distribution (both instrumentation and process air), potable water production using a reverse osmosis package and heavy fuel oil distribution.



ROM
Bin
Gyratory
Crusher
Alternative design
Ore
Sorter
To
Tailings
Accept
Cone
Crusher
Ore
Sorter
Accept
To
Tailings
Crushed Ore
Stockpile
HPGR
Feed Bin
HPGR
Bacteria
Water
Binder
H_2SO_4
Agglomeration
Drum
Extracting Agent
Organic
Aqueous
Copper
Extraction Circuit
Stripping Agent
Organic
Aqueous
Copper Stripping
Circuit
Crystalliser
Centrifuge
Flash
Dryer
$CuSO_4.5H_2O$
Cu
Electrowinning
Water
Heap Leach
Pads
Extracting Agent
Stripping Agent
Organic
Aqueous
Molybdenum
Extraction Circuit
Organic
Aqueous
Molybdenum
Stripping Circuit
Crystalliser
Flash
Dryer
MoO_3
Centrifuge
Lime
Thickener
Filter
Precipitation
Oxide
Residue
Raw Water
Distribution
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Water
Process
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Tails to Dry
Stacking
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Active Pond
Barren Pond
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NOT FOR CONSTRUCTION
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RERFENCE DRAWING TITLE
MIDAS
METS ENGINEERING
DEEP SOUTH
Haib Heap Leach Project
Overall Process Flow Diagram
Option 3: Upgraded Heap-Leach
J3690-P-PFD-003
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17.5 Process Description – Option 4

17.5.1 Crushing and Ore Handling

Run of Mine (ROM) ore is transported by truck from the mine to the ROM stockpile area near the crushing plant. The material is transferred to a ROM bin, which feeds to a primary crusher. The primary crusher is a gyratory crusher suited to higher crushing capacities. The open side setting (OSS) is expected to be set at 127 mm with an assumed P_{80} of 127 mm to be produced. The output of the gyratory crusher is fed to a diverter chute via a primary conveyor. The diverter chute distributes the material into two streams that feed two cone crushers feed bin in parallel. The feed bin discharge goes to a cone crusher vibrating feeder then into the cone crushers. The cone crushers closed side setting (CSS) are 32 mm, with an expected product P_{80} of 32 mm. The cone crusher product is fed to a screen in which the oversize is recycled to the diverter chute feed bin whilst the undersize is conveyed to a crushed ore stockpile.

The stockpiled crushed ore is conveyed from the crushing to another stockpile at the processing plant by a long distance conveyor.

The stockpiled ore reclaimed via apron feeders and transferred by conveyor to the grinding circuit where it is evenly distributed into two streams by a diverter chute. The grinding circuit consists of two high pressure grinding rolls (HGPR) in parallel. The HPGRs are fed via apron feeders.

The HPGR target crush size is 5 mm. The product is in closed circuit with a single deck screen and produces two size fractions. The oversize material is recycled back to the diverter chute and the undersize fraction stream reports to agglomeration. HPGR introduces micro-cracking that improves leach kinetics, allowing for maximum metal extraction during the heap leach process.

17.5.2 Agglomeration Drum

Agglomeration improves the permeability of the heap and facilitates even acid flow without pooling and increasing the amount of oxygen available for reaction. Additionally, pre-wetting will reduce the losses of fines from the wind and increase the leaching kinetics of the ore. Heap leaching requires good percolation throughout the heap to ensure maximum metal recovery is realised. Clays and fine particles can hinder solution flow through the heap, and the ore is often agglomerated to overcome this issue.

The undersize particles from HPGR and the DMS floats are combined with binder, sulphuric acid and water to agglomerate the ore into clumps. It is considered essential to undergo agglomeration. The binder and is added to the agglomeration drum in powder form. Additionally, it is a requirement to add 98% sulphuric acid to pre-leach the chalcopyrite and reduce heap leach times.

17.5.3 Heap Leach

The ore will be stacked by inclined conveyor stackers, producing a heap. This is a preferred stacking method due to conveyor stacking being able to reduce ore segregation which allows for increased permeability. Due to the use of sulphuric acid the conveyor edges must be moulded, open edge belts will severely corrode. Additionally, it is preferable to splice the conveyor belt instead of using clips as it reduces spillage and belt stress.

Drip lines are used primarily in arid environments due to the substantially reduced evaporation in comparison to heap sprays. The drip lines are buried 10 cm to 50 cm beneath the surface of the heap to minimise evaporation. The irrigation rate will be approximately 4-8 $L/h/m^2$. The primary heap pad will be irrigated with solution from the intermediate leach solution (ILS) pond. The secondary and the wash heap pad will be irrigated with solution from the barren pond.

The pad will require a double liner (HDPE) to minimise any possible loss of liquid from liner punctures. Due to the high evaporation rate in the area and close proximity of a river, there is a strict need to minimise risk of the heap solution entering the environment.

Primary Heap

The primary heap will consist of fresh ore from the agglomeration drum that is stacked using conveyors and irrigated from the intermediate leaching solution (ILS) pond. The ILS pond will contain a low concentration leached solution from the secondary pad. The primary heap is leached for 120 days and the pregnant leach solution (PLS) from the primary heap is collected in the pregnant solution pond. The leached ore then becomes the secondary heap by rerouting the flow of the particular piping.

Secondary Heap

The secondary heap will be irrigated from the barren solution pond. The barren pond solution contains leftover metal sulphates from the solvent extraction raffinate. The ILS from the secondary heap is collected in the ILS pond after the ore is spent. The spent ore becomes the washing heap by rerouting the flow of particular piping.

Washing Heap

The washing heap will be irrigated with solution from the barren pond. This ore is washed with solution through drip irrigation periodically (can be conducted over several years). The solution from the heap is collected in the barren pond and used for leaching of the secondary heap.

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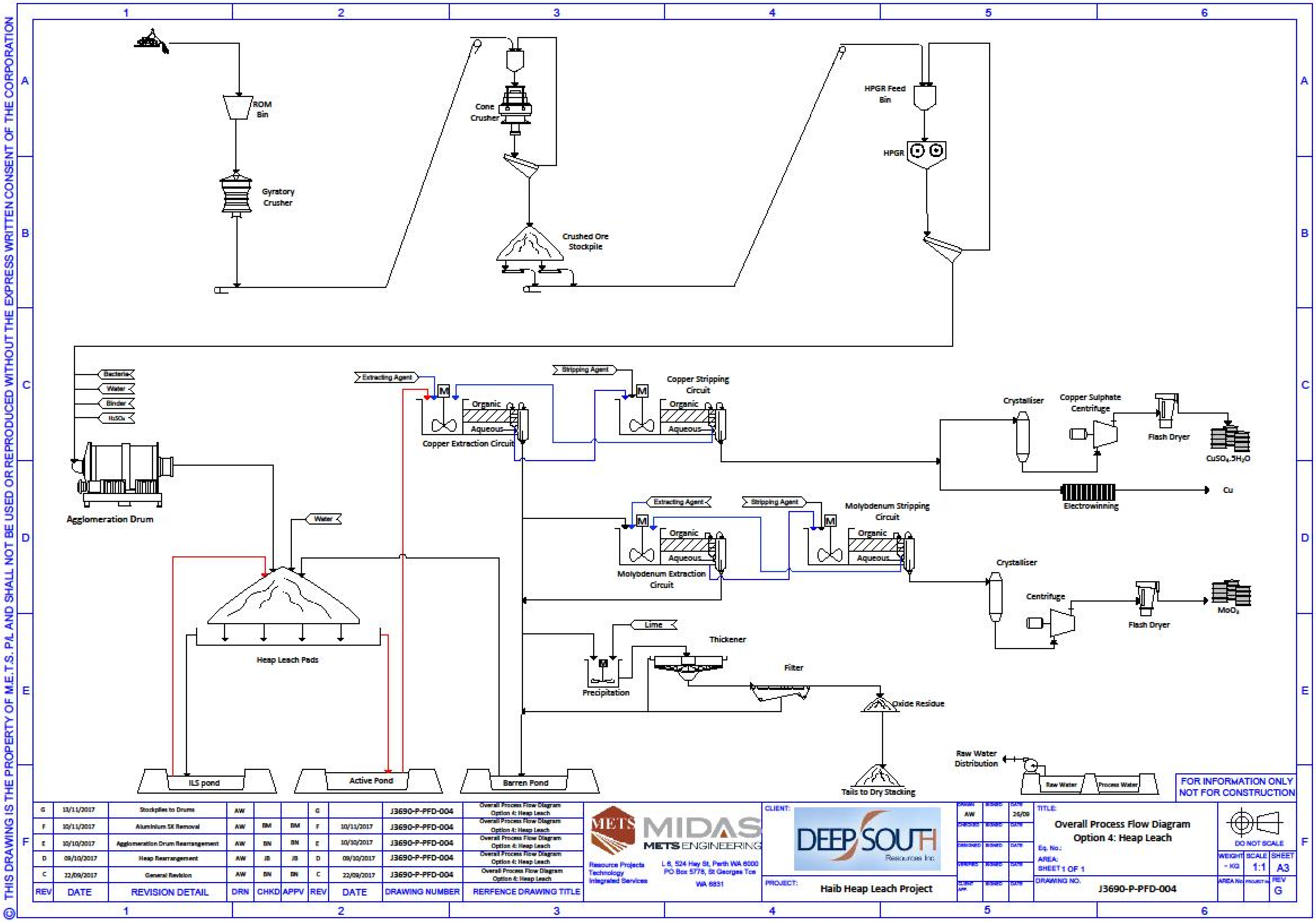
Water distribution covers the raw and process water dams. These will supply general plant water as well as a feed for potable water, fire water, gland seal water, reagents and cooling water.

17.5.9 Reagents

Reagents are mixed in an open area in covered tanks to prevent rain from damaging or reacting with the dry chemicals. The design incorporates accepted methods for mixing, holding, solution distribution and ventilation for each chemical according to MSDS and industry practice. Reagents are kept in a warehouse until they are required. Containment bunds and sump pumps are required for individual reagent handling areas. The sump pumps feed any spilled reagents into the respective tank depending on reagent area. The reagents area will provide storage and distribution for quicklime, flocculant, sulfuric acid, solvent extraction reagents and binder. The current design has an input of elemental sulphur which is burned to produce high concentration sulphuric acid.

17.5.10 Services

A services area will include air distribution (both instrumentation and process air), potable water production using a reverse osmosis package and heavy fuel oil distribution.


ROM Bin
Gyratory Crusher
Cone Crusher
Crushed Ore Stockpile
HPGR Feed Bin
HPGR
Bacteria
Water
Binder
H_2SO_4
Agglomeration Drum
Extracting Agent
Organic
Aqueous
Copper Extraction Circuit
Stripping Agent
Copper Stripping Circuit
Crystalliser
Copper Sulphate Centrifuge
Flash Dryer
$CuSO_4.5H_2O$
Electrowinning
Cu
Molybdenum Stripping Circuit
Molybdenum Extraction Circuit
Centrifuge
MoO_3
Water
Heap Leach Pads
Lime
Thickener
Precipitation
Filter
Oxide Residue
Tails to Dry Stacking
Raw Water Distribution
Raw Water
Process Water
ILS pond
Active Pond
Barren Pond
FOR INFORMATION ONLY
NOT FOR CONSTRUCTION
G 13/11/2017 Stockpiles to Drums AW G J3690-P-PFD-004 Overall Process Flow Diagram Option 4: Heap Leach
F 10/11/2017 Aluminium SX Removal AW BM BM F 10/11/2017 J3690-P-PFD-004 Overall Process Flow Diagram Option 4: Heap Leach
E 10/10/2017 Agglomeration Drum Rearrangement AW BN BN E 10/10/2017 J3690-P-PFD-004 Overall Process Flow Diagram Option 4: Heap Leach
D 09/10/2017 Heap Rearrangement AW JB JB D 09/10/2017 J3690-P-PFD-004 Overall Process Flow Diagram Option 4: Heap Leach
C 22/09/2017 General Revision AW BN BN C 22/09/2017 J3690-P-PFD-004 Overall Process Flow Diagram Option 4: Heap Leach
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Resource Projects Technology Integrated Services
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CLIENT:
DEEP SOUTH Resources Inc
PROJECT: Haib Heap Leach Project
AW 26/09
TITLE: Overall Process Flow Diagram Option 4: Heap Leach
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18. PROJECT INFRASTRUCTURE

18.1 Mine Area Power Requirements

The current Project site power requirements are estimated Table 18-1:

Table 18-1: Power requirement for each option.

	Installed Power (kW)	Power Draw (kW)
Option 1	21,215	17,285
Option 2	20,396	16,609
Option 3	19,315	15,282
Option 4	20,657	16,269

18.2 Mine Area Buildings

The pit mine site itself is located in a very rugged and steep area. Therefore, the cost of construction of the processing plant and heap leach pad nearby to the pit mine might be high. However, the mine area buildings required will depend on the processing option chosen.

Only the crushing plant will be constructed near to pit mine site for options 2 and 4. For options 1 and 3 the ore sorting will be part of crushing plant as shown in the figure below.

For options 1 and 3, ROM will be transported from the mine to the ROM stockpile area near the crushing plant. The material from the stockpile will feed the primary crusher. The product of the primary crusher is transferred to the ore sorting plant. The valuable mineral is transferred to the secondary crusher and then to the processing area. The gangue will be transferred to the reject stockpile located in an area close to the pit mine. This configuration will minimise the amount of material conveyed to the processing area.

Alternatively, for options 2 and 4, ROM will be transported from the mine to the ROM stockpile area near the crushing plant. The material from the stockpile will feed the crushing plant. The crushed ore is transferred to the processing plant. This configuration will increase the amount of material conveyed to processing area.

The processing area consists of the agglomeration plant, heap leach area, pond area, ore sorting plant, recovery plant, workshop and offices (Figure 18-1). It will be located in flat area at 4.5 km northwest of the mine. Thus, a conveyor of about 4.5 km length will be necessary to transfer the material to the processing area. It is foreseen that this will be a pipe conveyor to minimise dust losses.

The heap leach area will accommodate the primary, secondary and washing heap. The design of the heap leach pad is determined by various factors such as slope stability, seismic stability, amount of space available and climate. In the pond area are the pregnant leaching solution pond (PLS), the intermediate leaching solution pond (ILS), barren leaching solution pond (BLS) and the process water pond. The ore dressing plant consists of the DMS and flotation facilities (options 1 and 2). The metal recovery plant consists of the solvent extraction, electrowinning and crystallisation facilities.

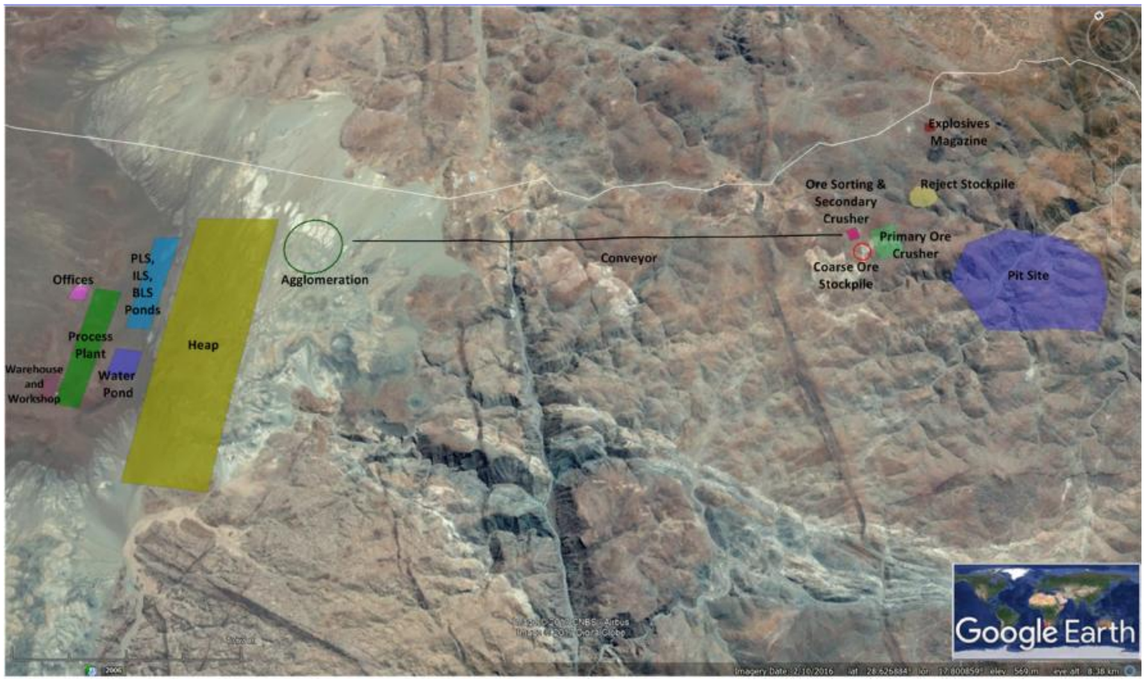

Figure 18-1: Mine site layout.

18.3 Explosives Storage

In Namibia, criteria apply to the possession and storage of explosives to ensure storing explosives without creating an unacceptable risk to the community and to the employees. Thus, a licence is required to possess and store explosives as prescribed by the Explosives Act 1956 and Regulations (GNR 1604 of 8 September 1972). Application for a licence shall be made to the chief inspector of explosives, who may issue such a licence subject to the observance of the regulations and after consultation with the local authority.

Design and location of a magazine for the storage of explosives will depend on the explosive category, quantity and distance to buildings such as railways, roads, dwelling-houses and navigable water. Table 18-2 specifies distances that shall form the basis on which applications for magazine for storage of explosives licences must follow.

Table 18-2: Distances requirements (in metres) to build a magazine for storage of explosives.

Net explosives	25- kilogram cartons	To other magazines			To railways, roads, open sports- ground, navigable water, or dwelling-house in same ownership as magazine and occupied by the owner or an employee			To other dwelling-houses or public buildings*		
Quantity kilograms	Number	Cat. X Mounded or un-mounded	Cat. Y mounded or un-mounded	Cat. Z or ZZ mounded	Cat. X mounded or un-mounded	Cat. Y mounded or un-mounded	Cat. Z or ZZ mounded	Cat. X mounded or un-mounded	Cat. Y mounded or un-mounded	Cat. Z or ZZ mounded
500	20	9	12	19	15	25	47	31	50	95
750	30	9	13	22	17	29	61	33	57	122
1 000	40	9	14	24	18	32	75	36	63	150
1 250	50	10	15	26	18	34	85	37	68	170
2 500	100	13	18	32	21	43	130	42	86	260
5 000	200	17	21	40	23	54	180	46	108	360
10 000	400	21	28	50	25	68	235	50	136	470
12 500	500	23	30	55	26	73	255	52	146	510
15000	600	24	33	58	27	78	270	54	156	540
20 000	800	25	37	65	28	85	300	55	170	600
25 000	1 000	26	40	70	29	90	320	57	180	640
30 000	1 200	27	45	75	30	100	345	60	200	690
40 000	1 600	27	50	80	30	110	380	60	220	760
50 000..........	2 000	27	55	85	30	115	400	60	230	800
75 000	3 000	27	65	100	32	135	470	65	270	940
100 000	4 000	27	75	110	33	145	510	65	290	1 020
150 000	6 000	27	90	125	35	170	590	70	340	1 180
200 000	8 000	27	95	135	35	180	640	70	360	1 280

Category X: Explosives having fire or slight explosion risk or both, with only local effect. Category Y: Explosives having mass fire risk, or moderate explosion risk, but not mass explosion risk. Category Z: Explosives having mass explosion risk with serious missile effect. Category ZZ: Explosives having mass explosion risk minor missile effect. Source: GNR 1604 of 8 September 1972, Namibia.

The Haib deposit has suitable areas to build a magazine since the surrounding area is unoccupied and the nearest settlement is 12km away from the Haib deposit.

18.4 Waste Dumps

Suitable and sufficient areas for tailings dams, recovery plant, waste dumps and heap leach pads are available within the EPL area but the chosen sites will be dependent on the eventual mine and plant design. The area of the property and surrounding remainder of the farm is State land and currently only used for emergency stock grazing purposes under lease from the State so mining will not conflict with any formal farming activities.

18.5 Tailings Waste Storage Facility

The function of a Tailings Waste Storage Facility ("TWSF") is to store tailings in an economic and safe manner in order to reduce environmental, community impacts and production costs. The construction of a TWSF must be adequate to the site and to type of tailings that will be produced. Due to the location of the Haib deposit, dry stacking of tailings will be conducted. Although the impact on capital and operating costs might be high, this method has advantages such as:

- Elimination of tailings failures.
- Easier to close and rehabilitate.
- Huge water conservation advantages.
- Smaller footprint.
- Good in areas of high seismic activity.

There are sufficient areas available to build a TWSF within the EPL area. However, supporting studies and analysis are required to choose its adequate location.

18.6 Power Transmission Line

The main north-south national power grid lines are some 85 km to the east of the Haib (Figure 18-2). Thus, an 85 km link and upgrade of the line be capacity would likely required should the project be developed.

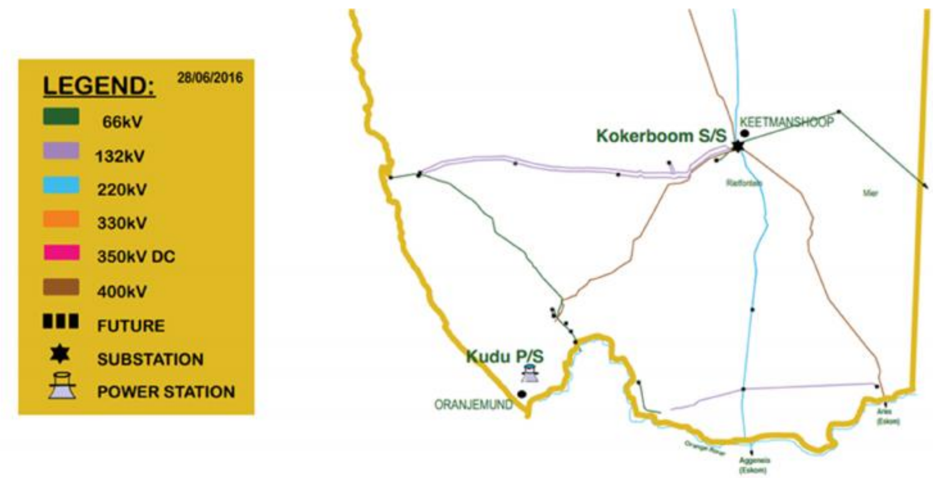

Figure 18-2: Power line transmission and substations in the south of Namibia. Source: Nampower annual report 2016.

18.7 Water

The Haib deposit straddles the Volstruis River (meaning the Ostrich river in Afrikaans), which is a tributary of the Haib River. Both are ephemeral tributaries of the Orange River which lies south of Haib.

The major water source is from the Orange River which is located about 15 kilometres by pipeline south of the main Haib deposit. However, due to the river being a shared resource between more than one country there are regulations that apply and future demand upstream may lessen the available water supply.

The Orange River is a deeply incised drainage with several nick-points. Haib lies below all of the main nick-points at a location where the Orange River elevation is approximately 200 metres above sea level.

The banks of the Orange River downstream of Vanderkloof Dam are heavily developed in many areas, principally for irrigation purposes. Both the Gariep and Vanderkloof dams are used to regulate the river flow for irrigation as well as to produce hydro-electricity during peak demand periods. Very little Orange River water is used for domestic or industrial purposes with the exception of that used in the Vaal River basin.

Very limited volumes of groundwater are available in the basement rocks of the southern Karas Region, since there are no productive aquifers. Lack of recharge and poor groundwater quality in most areas further aggravates the situation.

18.8 Water Management Pond

The Karas Region, where the Haib deposit is located, is an arid zone with low and erratic rainfall of about 50-100 mm/a, which can occur in the summer and winter seasons. Additionally, loss of water through evaporation only worsens the situation. Reliable water supply will therefore be critical for the successful and efficient operation of the mine.

Based on the evaluation of water for the project, 125-200 m^3/h (depending on the selected option) would be required. The key source of water will be the Orange River and the water recovered from tailings through the dry stacking process.

18.9 Telecommunication

Namibia has one of the most modern and sophisticated backbone infrastructures in Africa. Fibre optic cables are connected throughout the length of the country on the north-south and west to east axis. The countries telecommunications regulator is the Namibian Communications commission (NCC) working under the Namibian Communications Act of 1992. Telecom Namibia runs the largest Telecommunication network in Namibia.

A site telephone system will be used to connect together through various parts of the operation. Two-way radios will be used for communication between supervisors, mobile equipment operators, crusher operators and conveyor operators.

To facilitate the plant control system and communication between process areas, a wire network will be installed around the site.

18.10 Buildings

The project will require the development of the following infrastructure items in order to operate:

Table 18-3: Building required at Haib project.

Building	Description
Camps	Can be set up at either Noordoewer or Vioolsdrif as they are already established communities. These towns are 5 km apart and between them they have a hospital, a few luxury accommodations, petrol station, tradespersons, taxi services, Police station, medical clinic, border control
Crusher Control Room	Will provide a working space for engineers.
Reagent Shed	Will provide storage for reagents.
Canteen	Will provide area for cooking and dining facilities.
Metallurgical Laboratory	Laboratory to perform metallurgical testwork.
Assay Laboratory	Will provide laboratory equipment.
Open Area Storage	A fenced-off open storage area for equipment and materials that can be stored outside.
Maintenance/Warehouse	A facility will provide service the mobile equipment and for storage of equipment spares.
Control Room	Will provide working space for geology, engineering, and other operations support staff.
Office building	Will provide a working space for management, supervision.
Security Gate House	Will provide access control and security to the project.
Medical Centre	Will provide first aid services and emergency care.

18.11 Roads

Roads located near the deposit are well established and of sufficient quality (Figure 18-3). The deposit is located next to a main road that connects Namibia to South Africa, which is well maintained and suitable for large freight trucks. The road on the Namibian side is named Rundreise Namibia or state road B1 that extends from the North of Namibia at Oshikango to the South at Vioolsdrif. However, the only road construction required would be an upgrade to the existing 12 km long access road to site.

Figure 18-3: Roads close to Haib project. Source Google Maps, 2017.

18.12 Air Services

The airport of Oranjemund is located on the South West corner of the Namibian border at approximately 250 km from the deposit and has the appropriate services already established to transport the required personnel. The Keetmanshoop Airport located at 300 km from the deposit is the biggest airport in the Karas region in southern Namibia. It is situated five km outside the town of Keetmanshoop.

Additionally, there is the airport of Springbok in South Africa located at 157 km south of the deposit. Another option is the Kleizee Airport located in South Africa - its distance to the Haib area is 224 km.

18.13 Railways

The nearest railway station is located at the town of Grunau, some 120 km north along the main highway (Figure 18-4).

The area between the Haib and Grunau is almost completely flat and the local rail authority has confirmed that a link could be laid relatively easily; this would provide access to either the port of Luderitz or the port of Walvis Bay via Windhoek. Considering the available rail network in Namibia, the distance from Grunau to the port of Walvis Bay by rail is about 1200 km and 600 km to the port of Luderitz.

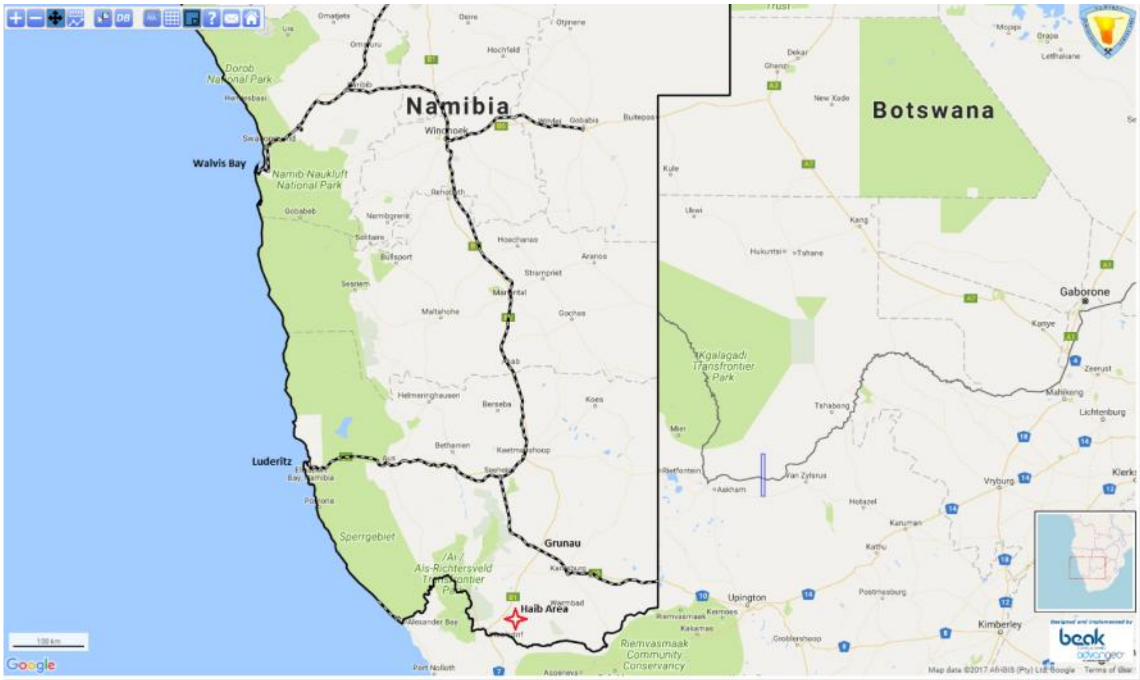

Figure 18-4: Railway network nearby to Haib Deposit showing the ports of Luderitz and Walvis bay.

18.14 Ports

Walvis Bay is Namibia's largest commercial port that is located approximately 1200 km away from Haib deposit. It is located half way down the coast of Namibia, with direct access to principal shipping routes. Walvis Bay is a natural gateway for international trade and is a sheltered deep-water harbour benefiting from a temperate climate. The long freight distance will incur significant costs for both import of raw materials and product export.

An alternative and preferable port that could be used is the port of Luderitz. It is located on the south-west coast of Namibia approximately 600 km away to Haib deposit. Traditionally, Lüderitz has been a fishing port, serving the needs of the Namibian fishing industry at a national level. The port is also an important shore base for oil and gas drilling operations off the southern coast and has also catered for the needs of the offshore diamond industry.

The rail connection could provide access to either the port of Luderitz or to Walvis Bay via Windhoek.

19. MARKETING STUDIES AND CONTRACTS

19.1 Copper

Copper is the main product that will be obtained from the process which will exist in the form of chalcopyrite or chalcocite concentrate from flotation, copper metal from electrowinning and copper sulphate from crystallisation.

19.1.1 LME Copper

Copper is one of the most widely used metals on the planet. China, Europe and the USA are the main global consumers of Copper. Copper will be produced on the cathode of the electrowinning cell as pure sheets which will be a pure (99%) solid. Pure copper metal is used for a variety of purposes with the major purpose is electrical wiring due to its great electrical conductivity. Additionally, copper is used in many metal works in making alloys such as brass and bronze which are stronger and more corrosion resistant than pure copper. A copper price of US$3.00/lb was incorporated in this economic analysis.

19.1.2 Copper Sulphate

Copper sulphate will be sold as a blue powder when the crystals are crushed and dried. Copper Sulphate is used in multiple industries such as arts, mining, chemical, pharmaceutical, healthcare and agricultural. The biggest use is for farming as an herbicide or fungicide as it can be used to control fungus on grapes, melons and berries. Additionally it inhibits the growth of E-Coli. Other uses include analytical reagents, past use as an emetic and dyes. In the healthcare sector, it is used in sterilisers and disinfectants. Industrial usage could be in adhesives, building, chemical, textiles industries, etc. where it is used to manufacture products like insecticides, wood preservatives and paints.

The Asia-Pacific region is the biggest consumer of copper sulphate due to the presence of large agricultural and animal husbandry industries. Other major consumers are North and South America and Europe. The main importers are listed as the United States with one fifth of the total global import volumes followed by Australia, Indonesia and Netherlands.

High purity copper sulphate has a 25% premium price based on the copper content in the sulphate. At US$3.00/lb of copper and 25% copper in the copper sulphate pentahydrate, a price of US$0.95/lb copper sulphate pentahydrate was used. The copper sulphate production was capped at 50,000 tonnes of copper sulphate pentahydrate (32,000 tonnes of anhydrous copper sulphate). According to a recent market study published by IMARC, the global copper sulphate market is expected to be more than 400,000 tonnes per annum by 2022. At the proposed production cap of 32,000 tonnes of anhydrous copper sulphate equivalent, this would represent approximately an 8% market share.

19.1.3 Copper Concentrate

The paid costs for the concentrate and applied penalties have great effects on the process economics. Highly desired concentrates have high copper grades, are quite clean and have gold and silver credits. Copper concentrate produced at Haib for Option 1 and 2 is expected to be high grade chalcopyrite with copper grades of approximately 33%. Although the plant head grade is low, the multi-stage upgrade allows for low-mass-high-grade feed to the flotation circuit. This coupled with a re-grind circuit will enable a clean concentrate to be produced. As the annual tonnage of concentrate is small, it is envisioned that the concentrate will be offered to Southern African copper smelters (possibly Tsumeb) to blend with lower grade concentrates by taking advantage of the high copper grades. Copper concentrate sales have treatment charges (TC) and refining charges (RC). For the purposes of the financial evaluation, a treatment charge of US$100/t copper concentrate and a refining charge of US$0.10/lb contained copper. An assumption of 95% payable copper was used.

19.2 Molybdenum Trioxide

Molybdenum trioxide (MoO_3) is a green/yellow powder which is mainly used as an oxidation catalyst and as a raw material for molybdenum metal production. It is used as a co-catalyst for the industrial production of acrylonitrile by oxidation of propene and ammonia. No sale of molybdenum has been considered for the PEA, with operating and capital expenses excluded from the financial analysis. Molybdenum is expected to be included in the indicated resource in the future, which will result in the economics being re-evaluated.

20. ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL & COMMUNITY IMPACT

20.1 Baseline Study

A multidisciplinary site survey conducted prior to or in the initial stage of a joint operational deployment. The survey documents existing deployment area environmental conditions, determines the potential for present and past site contamination (e.g., hazardous substances, petroleum products, and derivatives), and identifies potential vulnerabilities (to include occupational and environmental health risks).

Surveys accomplished in conjunction with joint operational deployments that do not involve training or exercises (e.g., contingency operations) should be completed to the extent practicable consistent with operational requirements.

20.2 Environmental Management Plan

The following draft Environmental Management Plan (EMP) details the measures to be adopted to address identified impacts during the construction and operational phases of the Project. The EMP details:

- Environmental elements – the environmental aspects requiring management consideration;
- Potential impacts – potential impacts identified in the EIS;
- Performance objective – the target or strategy to be achieved through management;
- Management actions – the actions to be undertaken to achieve the performance objective, including any necessary approvals, applications, and consultation;
- Performance indicators – criteria against which the implementation of the actions and the level of achievement of the performance objectives will be measured;
- Monitoring – the intended monitoring program and the process of measuring actual performance;
- Responsibility –responsibility for carrying out each action is assigned to a relevant person/organisation;
- Reporting – the process and responsibility for reporting monitoring results; and
- Corrective action – the action to be implemented in the case of non-compliance and the person/organisation responsible for action.

20.3 Project Environmental Assessment

Environmental impact assessments (EIA) ensure that the environmental impacts of a development proposal are fully considered before it is implemented. An environmental impact assessment determines the type and severity of an activity's environmental impact and is a normal part of the regulatory approval process and good due diligence practice.

Environmental impact assessment capabilities include:

- Flora and vegetation assessment
- Fauna and related habitat assessment
- Site specific characteristics assessment (aspect and relationship to the surrounding area)
- Formulation of environmental management plans
- Liaison with relevant government authorities (Environmental Protection Authority, Department of Parks & Wildlife, Department of Environment Regulation, Commonwealth Department of Environment & Energy, Water Corporation, heritage & the arts, and other local government bodies)
- Advice on other specialist scientific expertise that may be required
- Documentation of the assessment in the format required by regulators which can be used as part of an environmental management plan.

20.4 Environmental issues

20.4.1 Dust

The company will incorporate dust mitigation strategies, within reason, to minimise the negative impact on the environment, on site personnel and the community.

Personnel will continually monitor the site for excessive dust and take appropriate action to minimise exposure and dispersion. Mitigation strategies include:

- Job execution in a manner that reduces dust production
- Provide dust suppression equipment where needed
- Monitor, assess and respond to on-site dust observations
- Ensure vehicles, mobile equipment and significant foot traffic are primarily kept to sealed/stabilised regions
- Awareness of the prevailing wind direction to populated areas and implementation of job schedule accordingly

20.4.2 Noise

The company will implement measures to reduce noise production beyond unacceptable levels to ensure the environment, personnel and the community are not negatively impacted. The company will always comply with noise regulations of the area the site is located. If the site situated at a close proximity to residential dwellings, the company will not conduct noise generating work outside of the specified hours for weekdays and weekends.

When performing work, the company will ensure the environment, on site personnel and the community are not adversely impacted by incorporating the following strategies:

- Hearing PPE for personnel located within areas of elevated noise
- Noise suppression systems on equipment generating significant or ongoing noise

- Speed regulations to limit the noise from vehicles
- Awareness of the prevailing wind direction to populated areas and implementation of job schedule accordingly

20.4.3 Spillages

The company will be responsible for the prompt response and clean-up of any spillages that occur on the controlled site. On-site personnel will be trained and advised of the location for spill kits, if applicable, and the swift alleviation of a spillage.

All spillages, their contents matter, size and response are to be treated as an on-site incident and are to be reported to the site manager.

20.4.4 Contamination

The company will actively implement measures to avoid contamination of foreign objects, whether harmful or not, to areas outside of the site boundaries. This will include utilising and performing the following:

- Avoid seepage of materials into groundwater
- Clean vehicles and mobile equipment that cross site boundaries on a regular basis

The company will also ensure contamination of certain materials, particularly chemicals, is localised within sections of the site. This is primarily applicable to the adequate storage of chemicals, which are to meet the requirements outlined on the MSDS. Where applicable, bunding will be in place to ensure the containment of particularly hazardous materials.

20.4.5 Process Waste

The company will control and correctly dispose of any waste produced during the site operations. Any process waste shall be disposed of in accordance with statutory requirements. If waste materials are not suitable for disposal, the company will utilise treatment processes to ensure safe disposal, or will alternatively send the waste to a licenced facility for subsequent treatment and disposal.

Process waste disposal will meet local Government and other statutory bodies' requirements. In order to minimise process waste, the company will ensure the design and management of operational tasks is such that

20.4.6 Domestic/Municipal Waste

The company shall provide sufficient rubbish receptacles and industrial disposal bins for collection of waste and ensure that all such bins are emptied on a regular basis to prevent overfilling. Any hazardous substances shall be disposed of in accordance with statutory requirements at licensed facilities. All rubbish is to be placed in closed containers and not personnel should litter. The Site Supervisor will monitor the cleanliness of the site and take appropriate action if necessary. Personnel must actively seek to minimise rubbish and waste on site.

21. CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital Cost Estimate

21.1.1 Scope and Methodology

METS estimated capital costs for crushing, screening, grinding, floating, heap loading, leaching, solvent extraction and refining. It was assumed that the mining would be executed via contract mining and all associated capital was to be included in the mining operating expense. The estimates were made for a plant capable of crushing 8.5 Mtpa and were made for individual options including option:

- Option 1: Ore sorting upgrading, dense media upgrading, flotation and heap leaching of the tails
- Option 2: Two-stage dense media upgrading, flotation and heap leaching of the tails
- Option 3: Ore sorting upgrading and heap leaching of the upgraded material
- Option 4: Whole ore heap leaching

21.1.2 Basis of the Estimate

The scoping level study capital cost estimates are based on using historical equipment pricing and then factoring the materials and installation costs along with using the appropriate scaling factors. Vendors were contacted for major equipment such as crushers, HPGRs and ore sorters to obtain budgetary estimates. These quotes were scaled for options which had different throughput rates to the quoted amount.

Direct Costs

All direct equipment and infrastructure costs will be assumed to be new for this estimate and no second hand purchases are included. The cost of this equipment was estimated based on historical cost data collected by METS engineering and the installation costs factored to include costs for the following:

Earthworks

- Clearing of the site of vegetation
- Grubbing of roots and other materials from the site
- Bulk Earthworks
- Initial grading of the site for construction
- Major excavation (by machine) for concrete foundations
- Major backfilling (by machine) for concrete foundations
- Final grading and drainage contouring of the site
- Paving

Concrete

- Final trimming of the excavations

- Supplying and setting of formwork and shoring
- Supplying and installing reinforcing steel
- Supplying and installing embedded items
- Supplying and placing mixed concrete
- Finishing of the concrete
- Curing of the concrete
- Stripping of the formwork and shoring
- Final patching and finish
- Protective coatings for concrete surfaces
- Supplying and installing pre-cast concrete
- Supplying and installing concrete masonry

Structural Steel

- Detailing of structural steel from engineers drawings
- Supply and fabrication of steel materials and their fastenings
- Dismantling and salvage of steel materials
- Sandblasting and painting as required
- Transporting steel to site
- Unloading and “shaking-out” of steel in laydown areas
- Transporting steel to erection areas
- Checking the concrete dimensions before erection
- Erecting structural steel
- Plumbing and alignment of erected steel structures
- Tightening of all bolts according to specification
- Installation of metal roof and wall sheeting
- Installation of all ventilators and louvers
- Installation of doors and windows including frames
- Installation of flashing, edge strips, and sealers
- Installation of gutters and downspouts

Equipment

- Furnishing of the equipment by vendors
- Dismantling and salvaging equipment
- Transporting the equipment to site
- Unloading and storing on site
- Installing the equipment
- Mechanical testing of the equipment prior to start-up
- Sole plates, anchor bolts, safety guards, and all other items necessary to make the equipment operable

Piping

- Furnishing all pipe, valves and fittings
- Fabricating all pipe in a shop or on site
- Installing all pipe, valves and fittings
- Installing pipeline bodies for instruments
- Installing instrument airlines to final block valve
- Cleaning of the pipelines as specified
- Testing the pipelines as specified

Electrical and Instrumentation

- Installing all electrical equipment
- Installing all pull boxes, junction boxes etc.
- Installing all electrical cable and wire
- Furnishing all electrical equipment and bulk materials
- Dismantling and salvaging electrical equipment
- Installing all cable tray and conduit
- Furnishing and installing all hangers and supports
- Connecting all terminations
- Testing of all circuits and high voltage splices
- Furnishing all instruments at site
- Bench testing and calibration of all instruments as required prior to installation
- Furnishing and installing all supports and hangers
- Installing all pipe in-line instruments in pipeline bodies
- Installing all instrument airlines from block valve to instrument
- Installing all wiring between controllers, instruments, instrument blocks, power sources, and sending units
- Testing of all instruments, interlocks etc. after installation

Indirect Costs

As the costing is a Class 5 estimate, all indirect costs were calculated by factoring from the direct costs. The indirect costs include:

Engineering and Procurement

- Revising the Mission engineering drawings to accommodate the revised elevations and coordinates.
- Performing engineering on new equipment and associated equipment
- Planning, prioritizing and coordinating the engineering work
- Review or various trade off studies to minimize installation costs
- Review and finalization of the design criteria
- Review and finalization of the process flow sheet drawings
- Development of all process calculations
- Preparation of the Water Balance
- Preparation of the Material Balance
- Final sizing of all new equipment
- Development of the Equipment List
- Preparation of the Piping and Instrument Diagrams (P&IDs)
- Review of existing drawings
- Site visits as required
- Meetings as required
- Checking and collecting on-site dimensions
- Coordinate and evaluate geotechnical studies and reports
- Surveying
- Preparation of the General Arrangement Drawings
- Preparation of Detail Engineering drawings
- Preparation of all Civil and Site drawings
- Preparation of Electrical cable and conduit drawings
- Preparation of all Instrumentation layout drawings

- All other drawings required to provide a complete engineering design
- Preparation of specifications for new equipment
- Preparation of Requests for Quotation (RFQs)
- Preparation of contractor bid documents
- Evaluation of all bids
- Recommendations for all bids
- Preparation of the contract or purchase order documents
- Processing all change orders to contracts and purchase orders
- Preparation of the project schedule
- Preparation of the operating cost estimate
- Preparation of the capital cost estimate
- Provision of technical assistance during construction
- Provision of changes to the design during construction
- Management and administration of the engineering work
- Travel, communications, living cost, supplies, computers and all other costs necessary to engineer and procure for the project

Construction Management

- Coordination of the overall safety program
- Coordination of the construction work around the operation schedule
- Planning, coordination, and organization of the construction work with the contractors
- Construction surveying and survey control
- Inspection of the quality and progress of the work
- Surveying the work for correctness and quantities installed
- Approval/disapproval of all progress reports submitted for payment
- Identify potential problem areas and recommend solutions
- Review and approve/disapprove of change order requests
- Provision of quality testing, control and assurance of the work
- Provision of coordination and progress meetings with contractors and
- vendors
- Provision of all engineering documents to contractors
- Coordination of all engineering changes
- Provision of technical assistance as required
- Maintaining records of actual on-site installation
- Preparation of the As-built drawings
- Administration of the construction contracts
- Controlling and reporting of the project cost and schedule
- Approving and processing of all invoices
- Expediting, inspection and receipt of all deliveries

Field Office

- Provision of offices for contractor administration
- Provision of warehouse areas
- Provision of outdoor storage areas
- Provision of all utilities and infrastructure (roads, electrical, water, sewage, telephone, etc.) associated with the above
- Provision for control of the contractors ingress and egress

21.1.3 Summary of Capital Costs

A summary of the direct and indirect capital costs for each option assessed can be seen in Table 21-1. Option 3 produced the lowest capital expense due to the ore sorter reducing the downstream equipment size.

Table 21-1: Capital cost summary.

Cost (US$M)	Option 1	Option 2	Option 3	Option 4
Crushing & HPGR	54.3	50.0	56.2	53.3
DMS & Grinding	13.7	23.4	-	-
Flotation	3.0	3.3	-	-
Agglomeration & Heap Leaching	12.5	14.7	12.4	20.4
Copper Recovery	31.6	35.1	32.1	38.0
Iron Removal	1.6	1.9	1.8	2.7
Tailings	1.7	2.2	-	-
Water	2.9	3.7	2.8	2.8
Reagents	2.7	2.0	1.6	1.9
Services	2.0	2.0	2.0	2.0
Sulphuric Acid Production	21.4	27.1	22.0	27.3
Supporting Infrastructure	2.8	2.8	2.8	2.8
First Fill	10.8	13.0	6.0	8.3
Working Capital	16.1	18.1	14.0	16.0
Insurance	3.8	5.4	3.3	4.8
EPCM	16.1	18.1	14.0	16.0
Contingency	16.1	18.1	14.0	16.0
Commissioning	3.2	3.6	2.8	3.2
Accommodation & Temp Services	3.2	3.6	2.8	3.2
Spares & Tools	1.7	2.0	1.5	1.7
Total (US$M)	221.2	250.1	191.8	220.3

21.1.4 Accuracy Assessment

At a scoping study level the accuracy is assumed to be at ±30% of the CAPEX.

21.2 Operating Cost Estimate

21.2.1 Scope and Methodology

METS estimated operating costs for crushing, screening, grinding, floating, heap loading, leaching, solvent extraction, electrowinning and crystallisation. The estimates were made for a plant capable of crushing 8.5 Mtpa and were made for individual options including option:

- Option 1: Ore sorting upgrading, dense media upgrading, flotation and heap leaching of the tails
- Option 2: Two-stage dense media upgrading, flotation and heap leaching of the tails
- Option 3: Ore sorting upgrading and heap leaching of the upgraded material
- Option 4: Whole ore heap leaching

21.2.2 Cost Breakdown Structure

Table 21-2 to Table 21-5 outlines the operating cost structure for the four options that were assessed. As previously mentioned, the cost of mining was assumed equal to that of a similar Namibian project.

Table 21-2: Option 1 operating cost breakdown.

Area	Annual Cost (‘000 USD)	Unit Cost (USD/t ROM)	Unit Cost (USD/lb CuEq)
Mining	19,210	2.26	0.40
Processing	42,587	5.01	0.88
Product Freight	2,471	0.29	0.05
Wharfage & Shiploading	275	0.03	0.01
Administration	1,700	0.20	0.04
Royalty	4,251	0.50	0.09
Total	70,494	8.29	1.46

Table 21-3: Option 2 operating cost breakdown.

Area	Annual Cost (‘000 USD)	Unit Cost (USD/t ROM)	Unit Cost (USD/lb CuEq)
Mining	19,210	2.26	0.37
Processing	47,283	5.56	0.92
Product Freight	2,718	0.32	0.05
Wharfage & Shiploading	302	0.04	0.01
Administration	1,700	0.20	0.03
Royalty	4,523	0.53	0.09
Total	75,736	8.91	1.47

Table 21-4: Option 3 operating cost breakdown.

Area	Annual Cost ('000 USD)	Unit Cost (USD/t ROM)	Unit Cost (USD/lb CuEq)
Mining	19,210	2.26	0.41
Processing	38,696	4.55	0.82
Product Freight	2,109	0.25	0.04
Wharfage & Shiploading	234	0.03	0.00
Administration	1,700	0.20	0.04
Royalty	4,224	0.50	0.09
Total	66,173	7.79	1.41

Table 21-5: Option 4 operating cost breakdown.

Area	Annual Cost ('000 USD)	Unit Cost (USD/t ROM)	Unit Cost (USD/lb CuEq)
Mining	19,210	2.26	0.37
Processing	42,419	5.99	0.83
Product Freight	2,305	0.27	0.04
Wharfage & Shiploading	256	0.03	0.00
Administration	1,700	0.20	0.03
Royalty	4,617	0.54	0.09
Total	70,507	8.29	1.37

21.2.3 Basis of Estimate

Process operating costs were estimated by METS engineering using the equipment list generated from the flowsheets, the manning requirements based on similar projects and from equipment vendors. The cost estimates cover crushing, screening, heap loading, leaching, solvent extraction and refining.

Estimated Labour Rates

Personnel requirements were assumed for each area. Namibian wages were used to estimate the total payroll. A 30.5% overhead was applied to the annual salary for each person.

Estimated Consumable Costs

Consumable costs are based on both quotes from vendors and spares prices from previous METS projects that, where applicable, have been converted from AUD to USD (1 $AUD = 0.75 $USD). The consumption rates are based on vendor information, past projects and METS experience.

Estimated Reagents Costs

The reagent costs have been estimated based on direct quotes from suppliers, past projects and from online sources such as Kemcore. All reagents costs are in USD with an allowance for delivery to site from the Luderitz port. The raw water price is assumed to be equivalent to the average 2016 mine tariff of N$10.09 corresponding to US$0.71/kL at the exchange rate utilised throughout the project. The diesel price has been taken as $ 0.74/L based on the Namibian diesel price on the 6/11/2017.

Estimated Power Cost

The power is assumed to be able to be taken from the grid from one of the surrounding towns. The 2017/18 tariff for large power used from NamPower of N$1.21/kWh (US$0.085/kWh) was incorporated for the study.

Estimated Maintenance Cost

The maintenance cost is estimated as a factor of the equipment capital expense for each process area. A larger portion of maintenance was allocated to Area 100 (crushing).

Accuracy Assessment

At a scoping study level the accuracy is assumed to be at ±30% of the OPEX.

22. ECONOMIC ANALYSIS

22.1 Introduction

The project economic assessment has been conducted by METS and is developed based on accurate and up-to-date information. The economic analysis includes the calculation of Net Present Value (NPV) on a pre-tax basis. The estimates assume that the production, cost targets, pricing and sales goals are achieved. Any deviation from those values affects the determination of NPV. The internal rate of return (IRR), payback period and other financial metrics were calculated to assist with determining the project's viability.

22.2 Macro-Economic Assumptions

22.2.1 Metal Price Assumptions

Table 22-1: Assumed pricing data.

Commodity	Units	Unit Value
LME copper	lb	US$3.00
Copper concentrate – payable copper	%	95%
Copper concentrate – treatment charges	t concentrate	US$100
Copper concentrate – refining charges	lb contained copper	US$0.10
Copper sulphate pentahydrate – premium	% contained copper	25%
Copper sulphate pentahydrate	lb	US$0.95

22.2.2 Royalties

The royalty for gold, copper, zinc and other base metals is 3% of the total revenue.
Taxes

The corporate tax for non-diamond mining in Namibia is 37.5% (as per the Chamber of Mines Namibia).

22.2.3 Financing

The economic analysis has been run on a basis of 100 percent equity financing.

22.2.4 Inflation

The economic analysis does not account for inflation.

22.2.5 Mining Costs

Mining costs have been assumed to be $2.26 USD/t based on a similar Namibian project. This is assumed conservative as the strip ratio is expected to be around 2:1.

22.2.6 Rail Freight

Rail freight has been set to $45 USD/tonne of products to send the products to Luderitz port. This is estimated via US$0.075/tkm and a 600 km freight distance.

22.2.7 Wharfage and Ship Loading

The wharfage and ship loading costs have been assumed to be $5 USD/tonne of products to account for port costs and shipping costs.

22.2.8 Discount Rate

A discount rate of 7.5% has been incorporated for the base case scenarios. The sensitivity analysis assessed step changes of 1.25%.

22.2.9 Exchange Rate

Where applicable, a Namibian dollar to US dollar exchange of 0.07 was incorporated. When estimating costs from quotes METS have received in Australian dollars an AUD to USD exchange rate of 0.75 was used. A CAD/USD exchange rate of 0.80 was used for conversion of NPV values.

22.3 Technical Assumptions

It is assumed that the project ramp up will be achieved over three years. Due to delayed leach extractions, the first year is assumed to achieve 25% of the design production, 75% in the second year and 100% by the third year.

22.4 Economic Outcomes – Option 1

The economic outcomes of Option 2 are summarised in Table 22-2.

Table 22-2: Option 1 financial metrics.

	$3.00/lb Cu Price	$3.30/lb Cu Price	$3.60/lb Cu Price
Throughput (Mtpa)	8.5		
Copper Recovery (%)	77.1		
CAPEX	US$221.2M		
Total Operating Expense[1]	US$1.46/lb CuEq	US$1.47/lb CuEq	US$1.47/lb CuEq
$NPV_{7.5\%,pre\text{-}tax}$	US$645.1M (CA$817.6M)	US$816.4M (CA$1,020.5M)	US$987.6M (CA$1,234.5M)
$IRR_{pre\text{-}tax}$	25.9%	30.1%	34.2%
Payback $Period_{pre\text{-}tax}$	5.0 years	4.2 years	3.7 years
$NPV_{7.5\%,post\text{-}tax}$	US$421.0M (CA$526.3M)	US$528.1M (CA$660.1M)	US$635.1M (CA$793.9M)
$IRR_{post\text{-}tax}$	20.0%	22.9%	25.6%
Payback $Period_{post\text{-}tax}$	6.7 years	5.7 years	5.0 years
LOM	55 years		

[1]Variable due to change in absolute royalty payment due to increased revenue

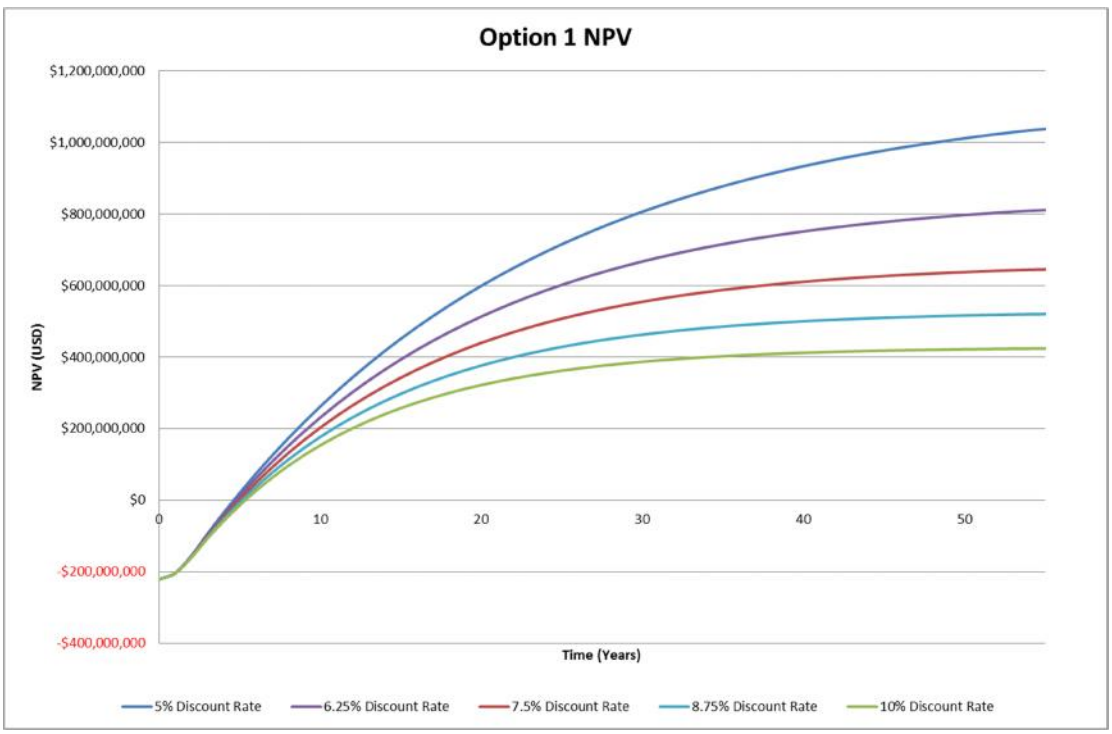

Figure 22-1: Option 1 NPV (pre-tax).

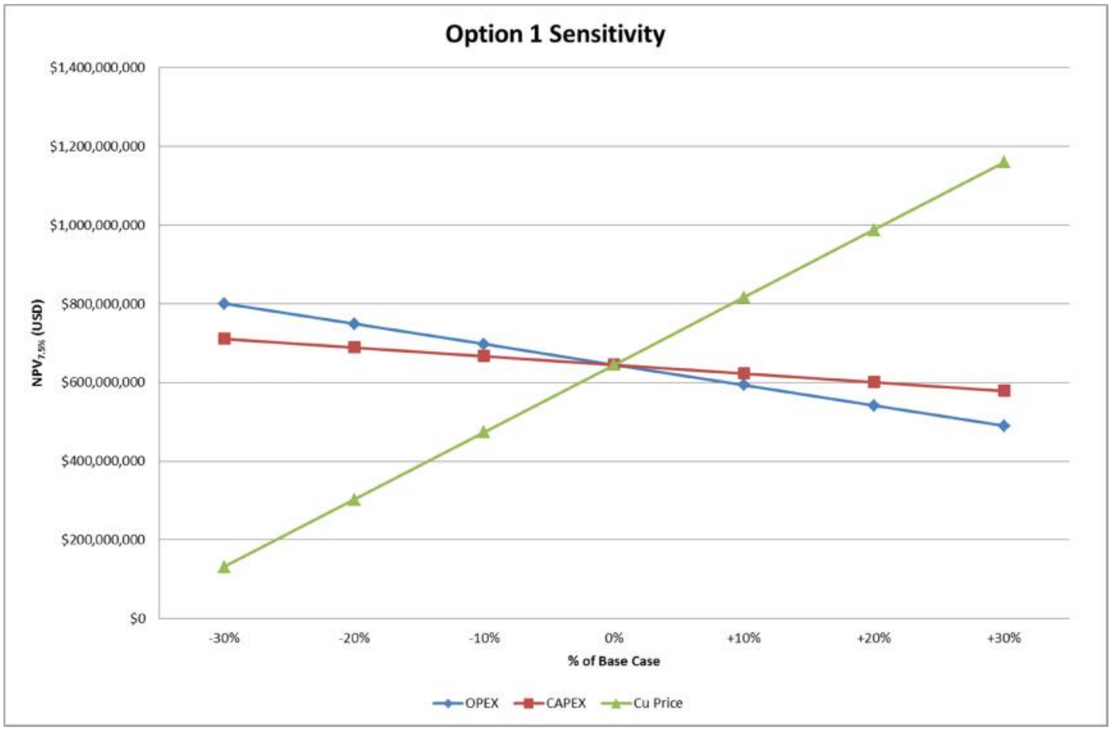

Figure 22-2: Option 1 sensitivity (pre-tax).

22.5 Economic Outcomes – Option 2

The economic outcomes of Option 2 are summarised in the table below.

Table 22-3: Option 2 financial metrics.

	\$3.00/lb Cu Price	\$3.30/lb Cu Price	\$3.60/lb Cu Price
Throughput (Mtpa)	8.5		
Copper Recovery (%)	82.1		
CAPEX	US\$250.1M		
Total Operating Expense[1]	US\$1.47/lb CuEq	US\$1.48/lb CuEq	US\$1.49/lb CuEq
$NPV_{7.5\%,pre\text{-}tax}$	US\$662.6M (CA\$828.3M)	US\$845.0M (CA\$1,056.3M)	US\$1,027.3M (CA\$1,284.1M)
$IRR_{pre\text{-}tax}$	24.4%	28.5%	32.4%
Payback $Period_{pre\text{-}tax}$	5.3 years	4.5 years	3.9 years
$NPV_{7.5\%,post\text{-}tax}$	US\$434.3M (CA\$542.9M)	US\$548.3M (CA\$685.4M)	US\$662.3M (CA\$827.9M)
$IRR_{post\text{-}tax}$	19.0%	21.7%	24.4%
Payback $Period_{post\text{-}tax}$	7.1 years	6.1 years	5.3 years
LOM	55 years		

[1]Variable due to change in absolute royalty payment due to increased revenue

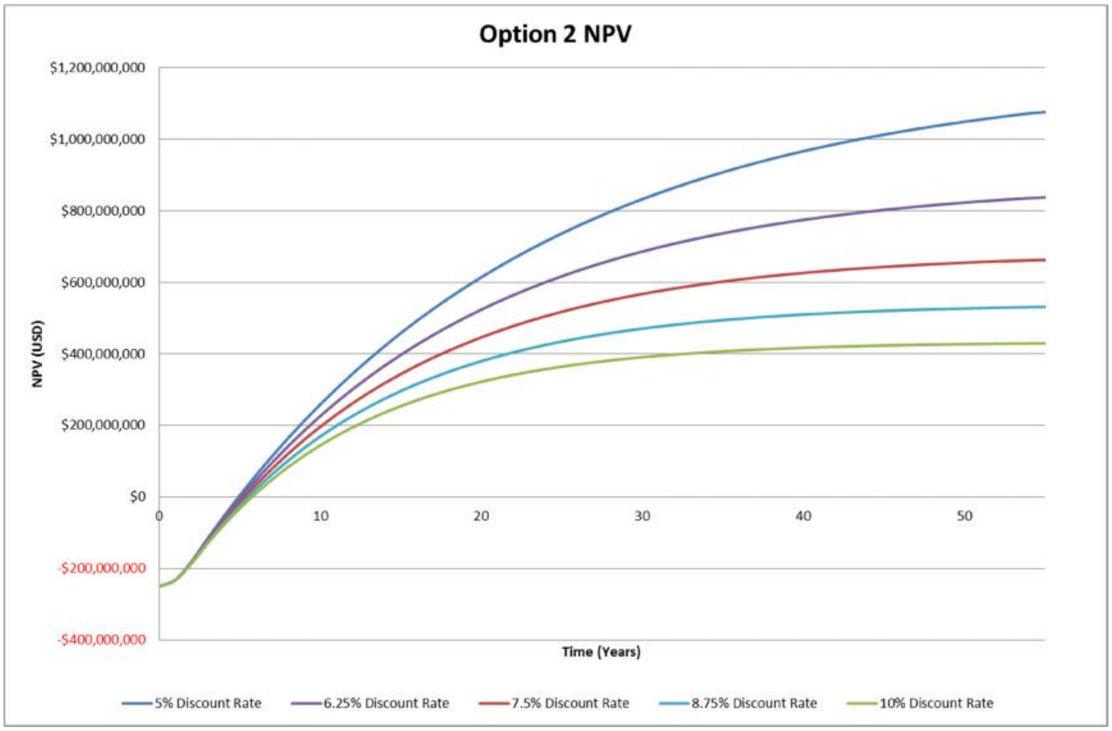

Figure 22-3: Option 2 NPV (pre-tax).

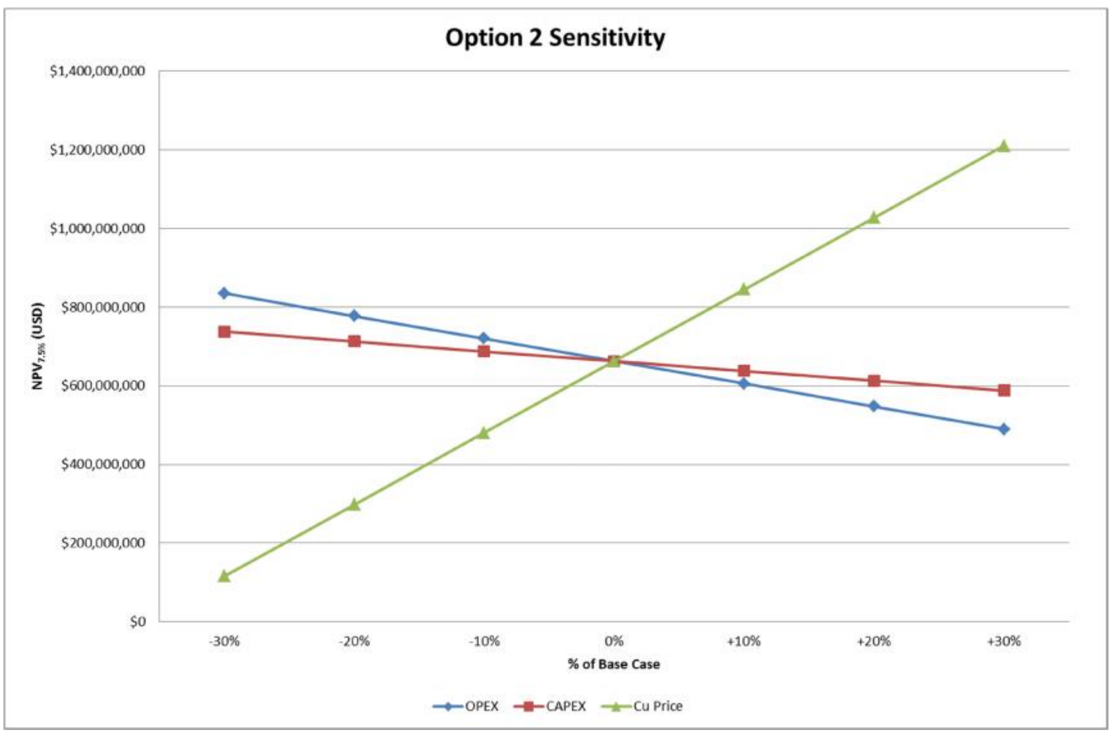

Figure 22-4: Option 2 sensitivity (pre-tax).

22.6 Economic Outcomes – Option 3

The economic outcomes of Option 3 are summarised in the table below.

Table 22-4: Option 3 financial metrics.

	$3.00/lb Cu Price	$3.30/lb Cu Price	$3.60/lb Cu Price
Throughput (Mtpa)	8.5		
Copper Recovery (%)	73.2		
CAPEX	US$191.8M		
Total Operating Expense[1]	US$1.41/lb CuEq	US$1.42/lb CuEq	US$1.43/lb CuEq
$NPV_{7.5\%,pre\text{-}tax}$	US$716.2M (CA$895.3M)	US$883.1M (CA$1,103.9M)	US$1049.3M (CA$1,311.6M)
$IRR_{pre\text{-}tax}$	30.4%	34.9%	39.2%
Payback $Period_{pre\text{-}tax}$	4.2 years	3.6 years	3.3 years
$NPV_{7.5\%,post\text{-}tax}$	US$463.1M (CA$578.9M)	US$567.4M (CA$709.3M)	US$671.3M (CA$839.1M)
$IRR_{post\text{-}tax}$	23.0%	26.1%	29.1%
Payback $Period_{post\text{-}tax}$	5.7 years	4.9 years	4.4 years
LOM	55 years		

[1]Variable due to change in absolute royalty payment due to increased revenue

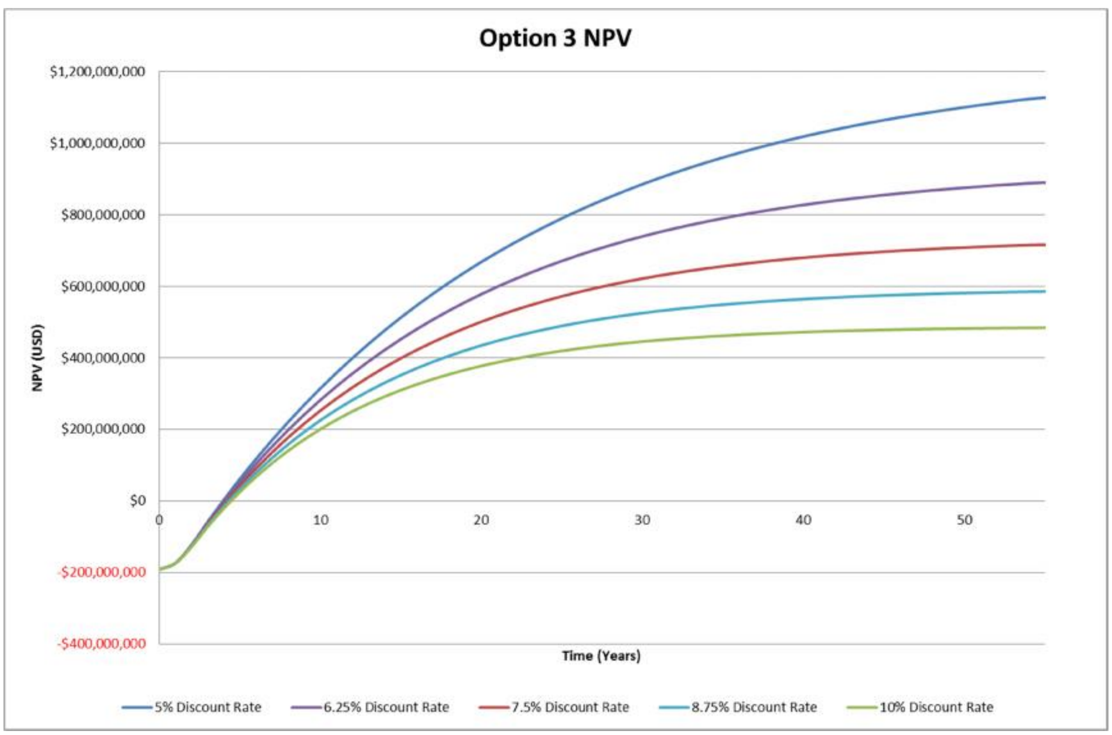

Figure 22-5: Option 3 NPV (pre-tax).

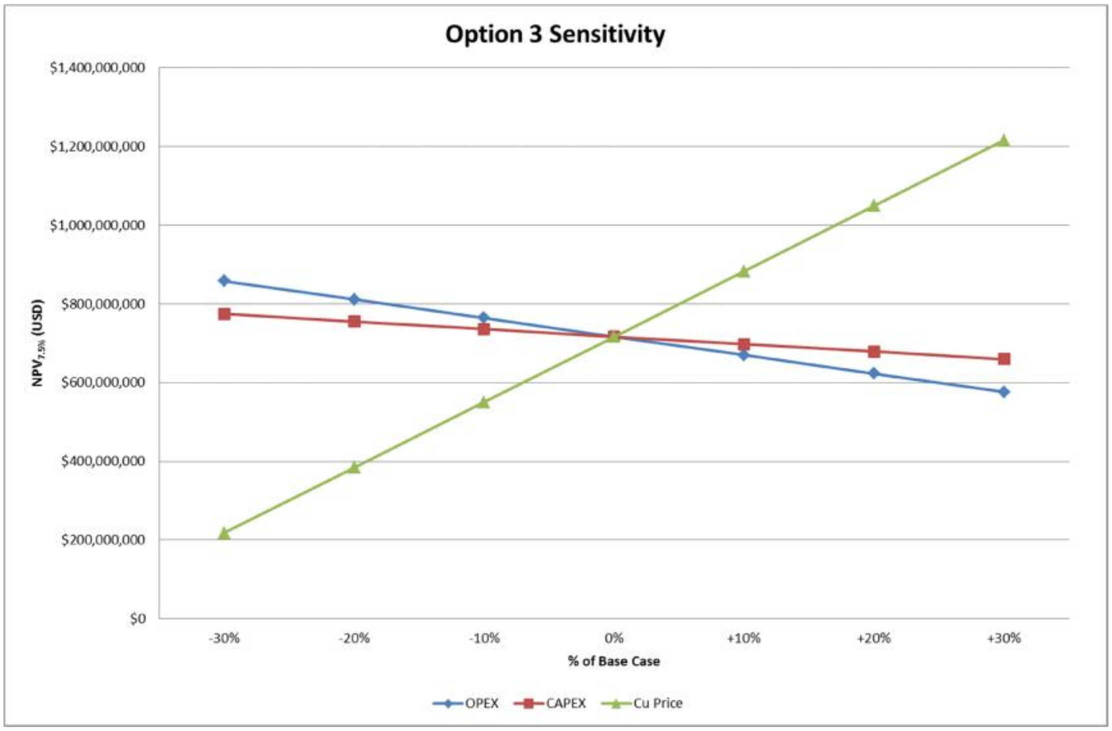

Figure 22-6: Option 3 sensitivity (pre-tax).

22.7 Economic Outcomes – Option 4

The economic outcomes of Option 4 are summarised in the table below.

Table 22-5: Option 4 financial metrics

	$3.00/lb Cu Price	$3.30/lb Cu Price	$3.60/lb Cu Price
Throughput (Mtpa)	8.5		
Copper Recovery (%)	80.0		
CAPEX	US$220.3M		
Total Operating Expense[1]	US$1.37/lb CuEq	US$1.38/lb CuEq	US$1.39/lb CuEq
$NPV_{7.5\%,pre\text{-}tax}$	US$794.1M (CA$992.6M)	US$976.5M (CA$1,220.6M)	US$1,158.1M (CA$1,447.6M)
$IRR_{pre\text{-}tax}$	29.7%	34.0%	38.2%
Payback $Period_{pre\text{-}tax}$	4.3 years	3.7 years	3.3 years
$NPV_{7.5\%,post\text{-}tax}$	US$514.1M (CA$642.6M)	US$628.1M (CA$785.1M)	US$741.6M (CA$927.0M)
$IRR_{post\text{-}tax}$	22.6%	25.5%	28.3%
Payback $Period_{post\text{-}tax}$	5.8 years	5.0 years	4.5 years
LOM	55 years		

[1]Variable due to change in absolute royalty payment due to increased revenue

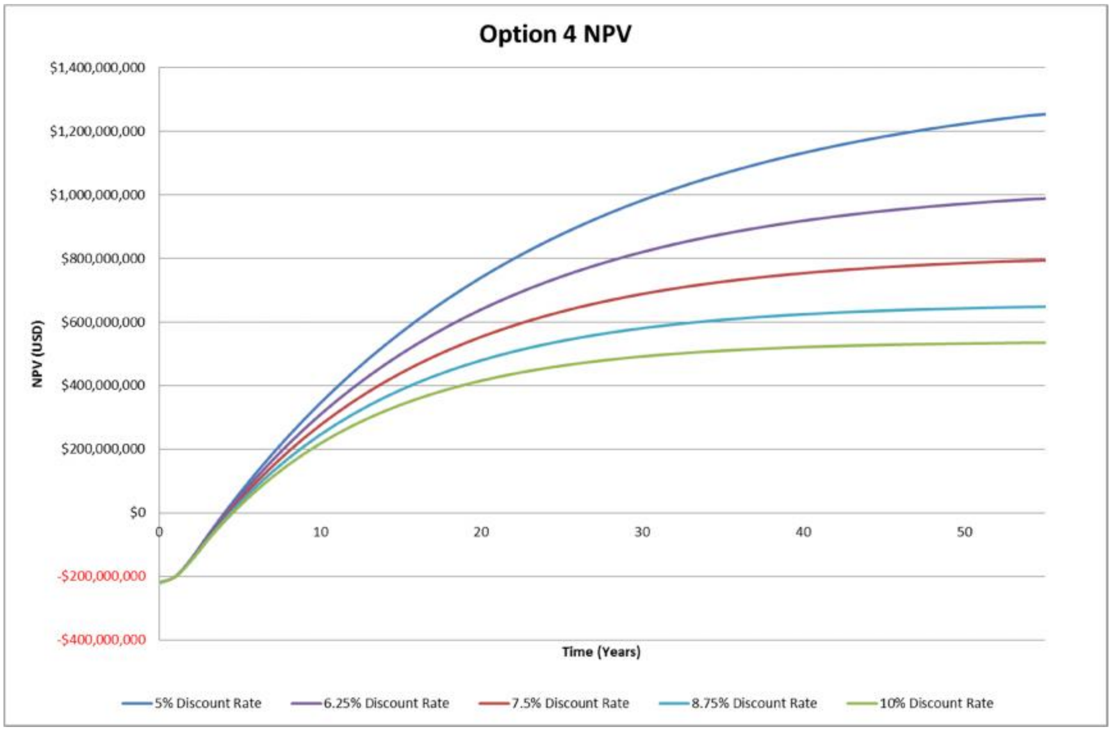

Figure 22-7: Option 4 NPV (pre-tax).

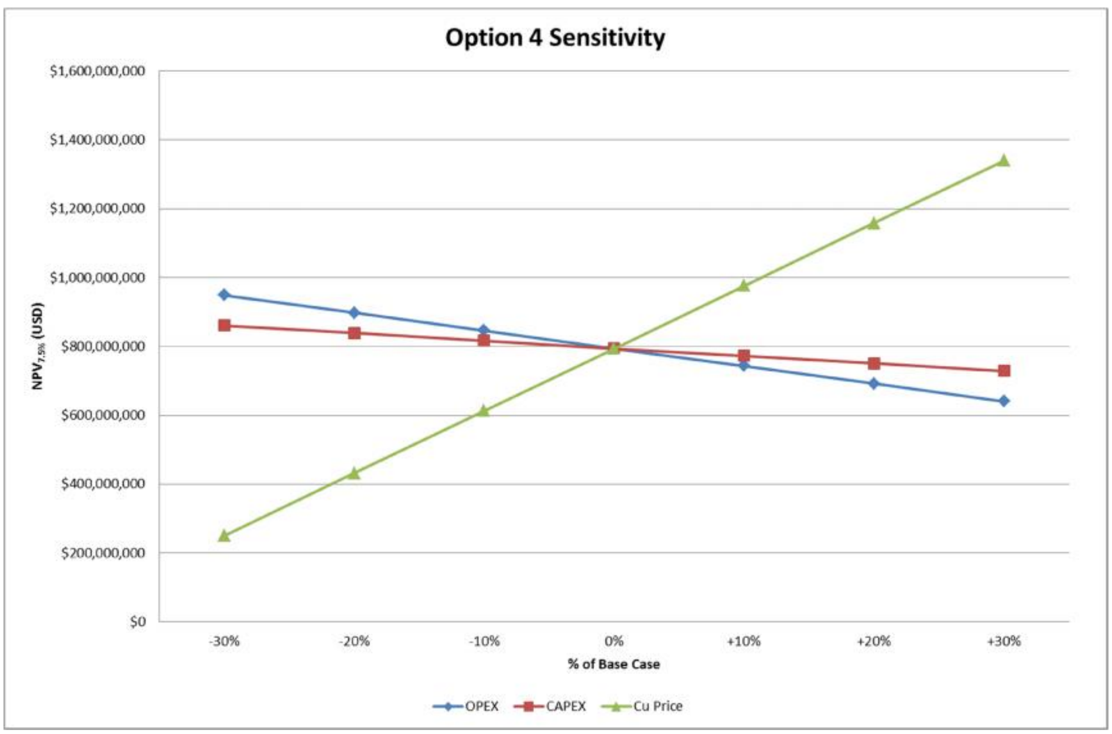

Figure 22-8: Option 4 sensitivity (pre-tax).

22.8 Economic Opportunity

As previously mentioned in the Recovery Methods, METS believes a ramp up to 20 Mtpa as the project is nearing positive cash flow will increase the financial viability. METS have developed a scenario which focuses on Option 3 – the best economic option in terms of IRR – in assessing the impact of increasing the scale of the project. The assessment looks at beginning the project at 20 Mtpa, however it is recommended to stage the expansion over a number of years (e.g. start at 5 Mtpa, increase to 10 Mtpa and then increase to 20 Mtpa for instance). The production of copper sulphate was capped at 50,000 tpa (hydrated basis) for this scenario. The capital cost has been scaled from the 8.5 Mtpa scenario. It is assumed that the unit operating cost is constant in terms of $/t ROM, however the unit cost per pound of copper equivalent changes due to the difference in equivalent copper due to the capped copper sulphate production. Table 22-6 outlines the key economic outcomes for the larger throughput scenario (using the base case figures – e.g. $3.00/lb copper price).

Table 22-6: Option 3 at an increased 20 Mtpa throughput.

	8.5 Mtpa Scenario ($3.00/lb Cu)	20 Mtpa Scenario ($3.00/lb Cu)
CAPEX	US$191.8M	US$320.5M
Total Operating Expense[1]	US$1.41/lb CuEq	US$1.44/lb CuEq
$NPV_{7.5\%,pre\text{-}tax}$	US$716.2M (CA$895.3M)	US$1,366.8M (CA$1,671M)
$IRR_{pre\text{-}tax}$	30.4%	38.6%
Payback $Period_{pre\text{-}tax}$	4.2 years	3.3 years
$NPV_{7.5\%,post\text{-}tax}$	US$463.1M (CA$578.9M)	US$854.9M (CA$1,061.9M)
$IRR_{post\text{-}tax}$	23.0%	28.6%
Payback $Period_{post\text{-}tax}$	5.7 years	4.5 years
LOM	55 years	24 years

[1]Higher unit operating cost for 20 Mtpa due to capped copper sulphate production (the equivalent copper pounds are not directly scaled due to the lower portion of the premium product).

A throughput optimisation study should be performed once a final process design has been selected.

22.9 Principal QP's Comments

Based on the findings of the economic analysis, the Haib project has significant potential to be a profitable project. Modern processing technology can be used to assist maximise the economic potential of such a large resource. Testwork validation would be required, although from the assumptions used the Haib project is economically viable.

23. ADJACENT PROPERTIES

There are several large properties currently held by other exploration companies that completely surround the Haib property. These are shown in Figure 23-1 below, which is a map extract from the Namibian Department of Mines and Energy [8] website. As far as I (Peter Walker) am aware, no comprehensive exploration programme for copper or base metal mineralisation has been reported on any of these properties despite historically reported visible surface indications of oxide copper, particularly to the southeast and east of Haib in Haib Volcanics and Vioolsdrif Intrusives. The larger, adjacent EPL's are tabulated below: -

Table 23-1. List of Properties and Owners Adjacent to the Haib Property [8]

EPL No.		
5327 36 540ha	Owner	Namibia Hua Yan Resources Explo & Devel Pty Ltd
	Tel. No.	+264-61-220465
	Minerals	Industrial Minerals & Precious Stones
	Granted	29-10-2013 expired 28-10-16 renewal pending
	Region	Karas
	District	Karasburg
5704 16 357.6ha	Owner	Desiree Rosilene Davids
	Tel. No.	+264 81 149 0504
	Minerals	Precious Stones
	Granted	28 January 2015 to 27 January 2018
	Region	Karas
	District	Karasburg
4182 44 819ha	Owner	Walenga, John
	Tel. No.	+264-61-262572
	Minerals	Base metals; rare metals; industrial minerals; precious metals; phosphate
	Granted	17-6-2014 renewal pending
	Region	Karas
	District	Karasburg
5187 9 369.6ha	Owner	Trevor James Rhode
	Tel. No.	+264 61 400 406
	Minerals	Precious Stones
	Granted	19 November 2014 – 18 November 2017 Renewal pending
	Region	Karas
	District	Karasburg
5783 44 492 ha	Owner	Giant Mineral Namibia cc
	Tel. No.	+264-81-244777
	Minerals	Base metals; rare metals; industrial minerals, nuclear fuels, phosphates.
	Granted	30 Sep 2014 application pending

	Region	Karas
	District	Karasburg
6406 19 378.4ha	Owner	Gemco Investments cc
	Tel. No.	+264 81 222 3445
	Minerals	Precious Stones
	Granted	14 July 2017 – 13 July 2020
	Region	Karas
	District	Karasburg
5804 3 741.7ha	Owner	Mickal Ngajozikue Tjituka
	Tel. No.	+264 61 26 2597
	Minerals	Base & Rare Minerals & Precious Stones
	Granted	3 June 2015-2 June 2018
	Region	Karas
	District	Karasburg
6415 2,784.5ha	Owner	Erikki Kanukwatange
	Tel. No.	+264 81 242 4348
	Minerals	Base & Rare Minerals & Precious Stones
	Granted	27 June 2017-26 June 2020
	Region	Karas
	District	Karasburg

N.B. The above licence details were correct at the date on the DME website of 4th December 2017.

Please note that EPL 5327 overlies the HM property EPL 3140 and has been granted for Industrial Minerals and Precious Stones. As far as the HM management and Directors know, there has been no exploration on this EPL (5327) and their application for renewal dated 28th September 2016 is still pending.

The remaining adjacent properties do not conflict in any way with the activities of HM.

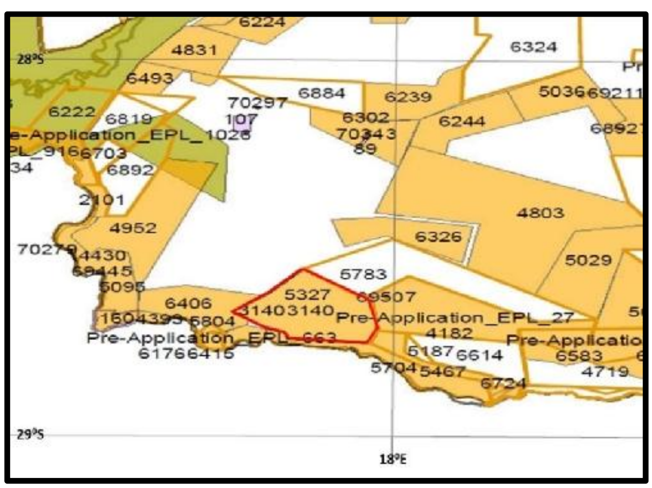

Figure 23-1: Location of Adjacent EPL's: A map, updated on 4 December 2017, extracted from the Department of Mines and Energy website (8) showing EPL 3140 (the Haib – outlined in red) and adjacent properties.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 General

Subsequent to the BD historic tonnage / grade estimation, NCJV completed a significant amount of work that was not reviewed by BD but does contribute significantly to the Haib knowledge base. This information will not be reviewed in detail here, however, a description and summary of this data is merited as it indicates the potential for future development of the Haib. Most of this data is available in the form of written reports and maps, some of which are available in electronic format.

Some of the additional data includes: -

- Data from the underground development of the adit. This involved some 150 metres of underground development of an adit at a nominal 2 x 2 metre cross-section with two short cross-cuts at the end of the adit (see photograph 2). This adit and cross-cut generated some 2,000 tonnes of fresh material for metallurgical test-work. The adit intersected higher grade material delineated by RTZ's close-spaced drilling on section 000E/W;

- Of the 2,000 tonnes of rock removed from the adit some 500 tonnes was sent to various laboratories for test-work. The balance is still stockpiled on site (see photograph 8);

- Geological mapping and sampling of the adit was carried out coincident with mining. Two sets of samples were collected for assay – the first being a sludge sample from the blast hole drilling, and the second being sidewall channel sampling. Assay samples were sent to the Scientific Services commercial laboratory in Cape Town, South Africa, and assayed for Cu oxide, Cu total, gold, silver, molybdenum, manganese and sulphur. The results of these analyses are available;

- Detailed surface geological mapping and drillhole re-logging was carried out. This included mapping of the Haib deposit and environs and the potential tailings site. This mapping included a geological re-logging of many of the old drillhole cores so that a geological model was developed. Much of this data is available. In addition, nearly all of the old RTZ and NCJV drill cores are still available on site at Haib (see photographs 3 and 4);

- 12 NQ drillholes were drilled by NCJV, totalling 4,306 metres. This programme was designed to complete in-fill drilling on the RTZ grid and to obtain some selective closer spaced drilling in the higher grade western end of the deposit;

- 5 x T2-101 large diameter holes, similar to PQ-size, were drilled totalling 627 metres; the drilling aimed at obtaining whole core samples of specific rock types for geotechnical testing. Existing NQ holes were twinned so that these rock types could be confidently sampled. These drillholes were also used to test grade variability over short distances. Analyses were as for the adit samples. Geotechnical logs were produced;

- An extensive structural mapping programme was carried out as a component of the open pit design. This study concentrated on the mineralised area and in part on areas where major infrastructure was to be located; this data is on record and available; and

- The area was flown for the production of ortho-photos and surface topographic maps at 1:10,000 scale over the mine site and 1: 30,000 scale over the entire prospecting licence area. Both ortho-photos and surface topographic maps are available. The ortho-photos were used by Teck geologists in their detailed geology mapping campaigns.

24.2 Contributing QP's Comment

Although the metallurgical testing and all of the environmental and rock competency tests and studies undertaken by the NCJV are still valid and are being used in the METS Preliminary Economic Assessment study, further bench-scale metallurgical work, geo-technical work prior to mine planning as well as assessments of infrastructure are required before further feasibility studies can be completed.

25. INTERPRETATION AND CONCLUSIONS

25.1 General Conclusions

The Haib mineralisation is undoubtedly a classic porphyry copper system and is probably one of the oldest known, preserved, porphyry deposits in the world [51].

Historical exploration work showed that the Haib project has large but low-grade copper mineralisation.

Teck's exploration results from the geological mapping, stream and soil sampling, geophysical survey and core drilling programmes to date contribute positively towards achieving HM's objective of providing a better understanding of the controls on high grade sections of the main Haib mineralised body and the nature of the satellite anomalies proximal to the main Haib body.

The deposit has been the subject of a new Resource Estimate using the results of historical drilling and also the newer Teck drilling and this study estimates that the Haib deposit has, using a 0.25% Cu cut-off grade, an in situ Indicated Resource of 457 million tonnes at an average grade of 0.31% Cu and an Inferred Resource of 342 million tonnes at an average grade of 0.29% Cu.

The Directors and majority shareholders of DSM and their wholly owned subsidiary HM are positive about the further development of the project and have commissioned a Preliminary Economic Assessment in order to show economic merit in development of the project towards full feasibility.

METS believes the low grade nature of the Haib porphyry copper deposit makes it an ideal candidate for heap leaching. Developments in heap leaching of refractory ores in conjunction with modern ore sorting technology have the ability to maximise the economic potential of the Haib project.

Based on the findings of the PEA, the Haib project has significant potential to be a profitable project. Modern processing technology can be used to assist maximise the economic potential of such a large resource. Testwork validation would be required, although from the assumptions used the Haib project is economically viable.

25.2 Significant Risks and Uncertainties

In addition to those outlined under Section 14 (Mineral Resource Estimates) the following significant risks and uncertainties can reasonably be expected to affect the reliability or confidence in the exploration information, mineral resource estimates and projected economic outcomes: -

- The risk that mineral rights may be cancelled for non-compliance with the conditions of grant. The exploration and surface rights held by HM are valid and HM have taken the appropriate steps in regard to meeting exploration spending commitments to ensure renewal of these rights at each renewal deadline [40]. I (Peter Walker) have relied for this conclusion on the certificates and a legal opinion letter [49] (see Appendices 1, 2 & 3) [42 & 50]. The Directors have provided me (Peter Walker) with a written assurance [65] that they are fully aware of their future obligations to report progress and abide by the agreed work programme within the time frames as per the Mining Act and as agreed with the Ministry of Mines and Energy in their acceptance of the conditions of grant. The Directors also assure me (Peter Walker) that there are no past or current disputes in relation to their legal title.

- The uncertainties regarding availability of existing infrastructure such as electric power, water, access routes, availability of trained personnel, transport and communications facilities. The location and proximity to existing, functioning infrastructure is highly favourable not only for exploration but also for further development of the property to a mining stage. In this regard, the availability of sufficient space for mining operations, processing plant, tailings and waste dump sites, heap leach pads and the highest and best use of the land as a mining property are positive factors.

- The geological understanding of the settings, lithologies and mineralisation controls for the target deposit type and knowledge of the regional geology is well enough understood to inform pre-feasibility study exploration programmes. It is the intention of HM to conduct further drilling for metallurgical and geo-technical purposes as and when required.

- The sampling methods employed by Teck meet or exceed industry standard best practice and the quality of both the exploration geochemical and drill core assay data is reliable and performed in accordance with exploration best practices and industry standards. The lack of historical drill assay certificates and QA / QC of the RTZ assay data may mean that HM will need to verify further RTZ drill core by an expanded re-assay programme; unless there is a large, verified assay database the risk of inaccurate estimation of grades for resource / reserve purposes is greatly increased. The over-estimation of grades by RTZ as shown by check assays completed to date are possibly caused by total Cu dissolution assay methods used by RTZ.

- Infill core drilling may be necessary where there are significant gaps in the model's database.

- There is a risk that resource estimates followed by economic studies may show that the project is not economic at lower copper prices and this may result in project failure.

- The political, economic, commodity market and technical risks and uncertainties which may affect the successful development of the property are adequately known and understood but future changes may impact substantially, in either positive or negative ways.

From a processing perspective, the biggest risks are considered to be:

- The current testwork does not reflect the proposed options. Additional testwork would be required to validate these options.

- There is uncertainty in regards to the supply of sulphuric acid to the site. The closure of nearby smelters increases the risk of a higher sulphuric acid price due to the increased transportation requirements. A study surrounding sulphuric acid and the possibility to have a sulphur burning plant should be assessed.

- The Luderitz Port may not have the facilities required to handle the volume of inward and outward goods. Further investigation into this is required once a selected process route is determined.

- Ore sorting required validation and variability. A chosen ore sorting route (XRT or MR) must be selected.

- If a long distance mine-to-plant conveyor is incorporated, the conveyor must have sufficient capacity to allow for project expansions.

25.3 QP's Comments

In my (Peter Walker's) opinion, HM is exploring a large volume porphyry copper deposit situated in an ideal location adjacent to modern infrastructure which has the potential to become a large copper resource. There already exists a significant body of technical data concerning the Haib mineralisation and the period between resource estimation, PEA studies, pre-feasibility and definitive feasibility studies could be relatively short.

In my opinion (Damian Connelly), the results from the PEA have been promising, and going forward METS recommends Deep-South Resources conduct a Pre-Feasibility Study (PFS). To improve confidence in the PFS results a more detailed metallurgical testwork will be required to validate the process options outlined in this report.

26. RECOMMENDATIONS

26.1 General

DSM has ownership of a significant, although highly challenging project in the Haib deposit which could rapidly progress to a feasibility study with a great deal of the investigative work already completed.

Teck Namibia correctly saw: -

- that the potential discovery of a satellite orebody could both increase the in-situ tonnage and provide a higher-grade zone that would help alleviate high initial CAPEX costs and,
- that the drill programme should also continue to investigate the potential higher-grade zones within the main Haib orebody.

These programmes have been continued for the last several years and were very successful in identifying satellite orebodies to the Haib and redefining the mineralization within the Haib main orebody. Unfortunately, to date, the satellite body exploration programme has not developed significant additional higher-grade zones.

The Teck geological mapping of the higher-grade zones indicated that these were insufficiently defined by the vertical RTZ drilling since many geological features controlling the distribution of mineralisation are sub-vertical meaning that some of the more important/ significant mineralized zones were under sampled. This suggests that there is an imperative to complete more infill drilling in the higher-grade zone identified by the new Resource modelling.

26.2 Pre-Feasibility Study

The results from the PEA have been promising, going forward METS recommends Deep-South Resources move to conduct a Pre-Feasibility Study (PFS) as the next phase of the project.

It is the objective of the Pre-Feasibility Study to complete all necessary work that is required in advance of the Feasibility Study/Basic Engineering Stage including evaluation of trade-off studies to determine the final project configuration. This is accomplished by completing and documenting the necessary trade-off studies with the objective to select a preferred project approach. Upon completion of this stage, the project team will have completely defined the project parameters and business criteria such that the strategic plan for project completion and implementation is fixed.

The objective is that the outcomes of a Pre-Feasibility Study for the Haib Project will support the following:

- An assessment of the likely technical and economic viability of the opportunity within a ±25% level of accuracy.

- Optimization of the different mining, process, location and project configurations to determine and recommend the preferred optimum to be engineered during the Pre-Feasibility Study.
- Evaluation of the project at different capacities.
- Determination of any fatal flaws in the opportunity.
- Development of the risk profile of the opportunity in relation to the key business drivers.
- Determination of the nature and extent of the Work Plan to complete further geological, mining, metallurgical, environmental and marketing work needed to be completed or undertaken during the Feasibility Study.
- An estimate of the costs, schedule and resources required to complete the Feasibility Study. In addition, an overall project schedule shall be prepared to indicate the overall timing of project implementation, commissioning and start-up, and ramp-up to full production.
- Identify resources (internal and external) and services required to undertake further work on the opportunity.
- If a pilot plant is required, it will be implemented during this stage.
- Upgrade the mineral resource (if required).
- Stakeholder considerations and plans.
- Risk assessment further refined and mitigation plans established.

26.3 Metallurgical Testwork

To improve confidence in the PFS results more detailed metallurgical testwork will be required. Most work to date has focussed on the potential of processing options and is not sufficient enough to truly evaluate their feasibility with confidence.

26.3.1 Drill Core

Drill core should be used in the next stage of testwork. The use of drill core will minimise the risk of drilling methods significantly changing the properties of the test material, providing high quality sample for the tests described here. The drill core will need to be representative of the resource and would preferably be fresh material. In order to achieve this, METS recommends establishing a drilling programme to cover both testwork samples as well as future exploration. The drilling programme will be needed to increase the level of confidence in the resource estimate.

26.3.2 Ore Sorting

An investigation into ore sorting is highly recommended for the next stage of the project given that the ore sorting option shows the most promise at the time of writing this report. An accurate assessment is required for ore sorting technology to prove that the assumptions made during the PEA were valid. The ore sorting testwork with MR technology will need to be conducted with CSIRO, the developers of the technology. XRT ore sorting testwork should be conducted as a comparison. The results from this testwork will provide sufficient detail for an accurate assessment of the feasibility of ore sorting technology for the Haib Project. In addition, a definitive conclusion will be drawn on which ore sorting technology is better suited to the project.

26.3.3 Column leach Tests

Column leach tests will be required for the PFS to determine several key heap leaching parameters with higher accuracy than those used for the PEA. Previous testwork was designed to test the amenability of the ore to leaching by sulphuric acid. The testwork revealed that copper can be leached from the ore, however, the conditions did not accurately represent the conditions that will be experienced in a heap leach. The results from a pilot scale heap leach operation will provide sulphuric acid consumption and metal recovery data for the economic assessment.

Column leach tests are small scale laboratory tests designed to evaluate the amenability of an ore to heap leaching. Like a full scale heap leach the ore is agglomerated, the agglomerated ore is placed in a column (avoiding packing of material) and a solution is percolated through the column with intermittent monitoring to determine acid consumption figures.

The PLS from the column leach tests can be used for solvent extraction testwork which will be required to determine the configuration of the solvent extraction circuits for molybdenum and copper. Optimisation of leaching and stripping solutions is crucial to optimisation of the solvent extraction circuit.

26.4 Engineering

Engineering design will be required at a sufficient level to evaluate the project within a $\pm$ 25% level of accuracy. Factors such as heap leach design will be critical to the success of the project and the design needs to be established early in the Pre-Feasibility Study. The engineering design from this stage of the project should also have enough detail to move into a feasibility study where the base designs can be further developed.

26.5 Mining Study

A mining study should be conducted during the Pre-Feasibility Study to assess in detail the feasibility of mining the ore for the Haib Project.

26.6 Marketing Study

A market study should be completed during the Pre-Feasibility Study, looking at the industry, a current market analysis, competition, future market potential, potential buyers and sources of revenue and sales projections.

26.7 Environmental

Environmental impact assessments will be required for obtaining approvals for initiation of the project if the Pre-Feasibility Study returns promising results.

26.8 Opportunities Study

Several opportunities have been identified for the Haib Project, all of which should be investigated in the future in order to maximise the project’s potential.

26.8.1 Solar Energy

Given the semi-arid climate of Namibia, a solar energy farm may be an option for reducing the unit cost of power. This will also have positive social impacts for the project, which is expected to have a long life.

26.8.2 Project Expansion

The resource tonnage allows for possible multiple expansion stages to be executed should the project proceed to production. A staged approach is recommended in order to de-risk the project by executing expansions once the operation nears positive cashflow.

26.8.3 Optimisation

During operation there should be ongoing optimisation studies to ensure the project financials are maximised. This should include optimisation of the metallurgy, recoveries, products and raw materials.

26.8.4 Sulphur Burning Plant

The design for each option as it stands involves the burning of sulphur to produce sulphuric acid. There are several possibilities for sulphuric acid sourcing, including purchasing from smelters within Namibia. Tsumeb has an off-gas cleaning facility that produces sulphuric acid for sale. An alternative would be Vedanta Resources who have suggested producing and sending zinc concentrate from their Gamsberg zinc mine to Skorpion mine, which is located closer to the Haib site than the alternative options. Buying in sulphuric acid at the start of the project life and building a sulphur burning plant once the project is cash flow positive may provide a better economic scenario. This will allow for the sulphur burning plant capital to be deferred and the payback period to be shortened. This trade-off study will have to be completed once accurate sulphuric acid pricing and the source of the acid have been obtained.

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QUALIFIED PERSON CERTIFICATE

I, Damian Edward Gerard Connelly, B.Sc. App Sc, FAusIMM, FIEAust, as principal author to this report titled "Haib Copper Project 2018 Preliminary Economic Assessment" (the "Report") with a report date of 28th February 2018, do hereby certify that:

1. I am an independent consulting metallurgist operating under the auspices of Mineral Engineering Technical Services, located at Level 3, 44 Parliament Place, West Perth, 6005, Australia. Tel: +61 (08) 9421 9000.

2. I graduated with a Bachelor of Science degree in Applied Science in 1973 from the University of Adelaide, in Australia.

3. I am a Professional Metallurgist registered as a Fellow of the Australasian Institute of Mining and Metallurgy and a Chartered Professional Engineer (met). I am also a Fellow of Engineers Australia.

4. I have worked as a metallurgist for a total of 45 years since my graduation from university.

5. I have worked as a consultant Metallurgical Consultant to the Mineral Processing Industry for the past 30 years, which has involved working on feasibility studies, detailed design, plant construction, due diligence work and more.

6. I have read the definition of a Qualified Person as set out in NI 43-101 as amended in 2011 and certify that by reason of my education, 45 years of relevant experience and professional affiliations, I fulfill the requirements to be a Qualified Person for the sections prepared by myself in the Report.

7. I am responsible for sections 16 to 22 and have contributed to sections 1, 2, 25 and 26.

8. I have visited the Haib site in 2006. The objective of the site visit was to assess the surrounding infrastructure, view drill core samples and obtain a general feel for the site.

9. As of the date of this certificate, I am not aware of any material fact or material change with respect to the subject matter of the Report, which is not reflected in the Report, the omission of which would make the report misleading.

10. In terms of section 1.5 of NI 43-101 “Standards of Disclosure for Mineral Properties” I am independent of the commissioning entities, being the Issuer, Deep-South Resources Inc., and its subsidiaries and associates Deep South Mining Company (Pty) Ltd and Haib Minerals (Pty) Ltd applying all of the standard tests of independence.

11. I have read NI 43-101 as amended on June 30, 2011 and confirm that this Technical Review Report has been prepared in compliance with the Standards and Guidelines as set out in that document.

DAMIAN E.G. CONNELLY B.App.Sc. FAusIMM. (CP) Met. FIEAust.
Dated: 28th February 2018.

QUALIFIED PERSON CERTIFICATE

I, Peter W.A. Walker, B.Sc. (Hons) Geology, M.B.A., Pr. Sci. Nat., as a contributor to this report titled “Haib Copper Project 2018 Preliminary Economic Assessment” with a report date of 28th February 2018, do hereby certify that:

1. I am an independent Consulting Geologist conducting work under the auspices of P&E Walker Consultancy cc of 41 Dennekamp, Main Road, Kenilworth 7708. Republic of South Africa. Tel: +27 (21) 762 1915 Cell: +27 (72) 411 1108 e-mail: elipet@mweb.co.za

2. I graduated with a Bachelor of Science (Hons.) degree in Geology in 1972 and an MBA in 1982, both from the University of Cape Town, South Africa.

3. I am a Professional Geologist registered with the South African Council for Natural Scientific Professions, registration No.400064/99;

4. I have worked as a geologist for a total of 40 years since my graduation from university. My relevant experience for the purposes of this Technical Report is:

- Seven years (1971 – 1978) as an exploration geologist in South Africa engaged in the mapping, drilling and evaluation of base metal deposits.
- Five years (1978 – 1982) as an exploration geologist in South Africa engaged in the exploration for Uranium and Tungsten deposits. During this period, I had mine visits to Climax Molybdenum mine amongst others in the USA, Australia, Canada and Brazil and also worked for three years on the discovery and evaluation of the Riviera porphyry Tungsten – Molybdenum deposit in the South-Western Cape, South Africa.
- Six years (1989 – 1995) as a senior exploration geologist in Namibia in the exploration, drilling and evaluation of gold and base metal deposits.
- Seven years (1995 – 2002) as exploration manager for first Trans Hex International Ltd and then Group exploration manager for Trans Hex Group, engaged in the valuation and assessment of new alluvial and kimberlite diamond projects, their exploration and management through to production.
- Three years as an independent, sole practitioner consultant (2002 – 2004) advising and writing competent person reports for exploration & mining companies.
- Thirteen years (2004 – present) as Principal of P&E Walker Consultancy cc, an independent geological consulting closed corporation engaged in advising and writing competent person reports for exploration and mining companies.

5. I have read the definition of a “Qualified Person” as set out in NI 43-101 as amended on June 30 2011, and certify that by reason of my education, 40-years of experience in exploration geology, mining, and affiliation with a professional

association I fulfill the requirements to be a "Qualified Person" for the purpose of preparing this Report.

6. I am responsible for sections 1 to 11, 13 & 15 to 19 and Annexures of this independent technical review, Resource Estimate and Preliminary Economic Assessment report.

7. I visited the Haib Project site described in this report on various occasions between 1989 and 1995 on Geological Society of Namibia field excursions and more recently on the 24th January 2012 and on the 30th June 2015. I have had no previous involvement with the Haib property.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the report contains all scientific and technical information that is required to be disclosed to make the report not misleading.

9. In terms of section 1.5 of NI 43-101 "Standards of Disclosure for Mineral Properties" I am independent of the commissioning entities, Deep South Mining (Pty) Ltd as well as of the Issuer, Deep South Resources Inc. their subsidiaries and associates applying all of the standard tests of independence; P&E Walker Consultancy cc is also independent of the commissioning entities and the Issuer, their directors, senior management and advisors.

10. I have read NI 43-101 as amended on June 30, 2011 and confirm that this Report has been prepared in compliance with the Standards and Guidelines as set out in that document.

P.W.A. WALKER *B.Sc. (Hons.) MBA Pr. Sci. Nat. FSEG MSAGS MGSN.*
Dated: 28th February 2018.

QUALIFIED PERSON CERTIFICATE

I, Dean Sean Richards, B.Sc. (Hons) Geology, Pr. Sci. Nat., as a contributor to this report titled "Haib Copper Project 2018 Preliminary Economic Assessment" (the "Report") with a report date of 28th February 2018, do hereby certify that:

1. I am an independent consulting geologist operating under the auspices of Obsidian Consulting Services cc, located at 46 Hamilton Ave., Craighall Park, 2196, Johannesburg, Republic of South Africa. Tel: +27 (11) 268 5772 Cell: +27 (82) 322 2466 e-mail: deanr@obsidianconsulting.co.za.

2. I graduated with a Bachelor of Science (Hons.) degree in Geology in 1991 from the University of Natal, Durban in South Africa.

3. I am a practicing professional geologist with 26 years of relevant experience and am registered with the South African Council for Natural Scientific Professions (reg. 400190/08) and a member of the Geological Society of South Africa.

4. I was responsible for the compilation of Sub-section 12.3 (Assessment of Quality Control Data) and Section 14 (Mineral Resource Estimate) of the Report.

5. I am a Qualified Person for these as that term is defined in the National Instrument 43-101 on the basis of the following relevant experience;

- Seven years (1992 – 1998) as an exploration geologist in South Africa engaged in mapping, drilling, logging, modelling and estimation of ferrous, base and industrial minerals projects.

- Eight years (1998 – 2006) as Technical Services Director of Gemcom Africa (Pty) Ltd responsible for Competent Persons Reports, 3D modelling, resource estimation, mine design and mine planning studies for diamond, precious metal, ferrous and base metal mines and exploration projects. Of particular relevance are:

    - Konkola Deeps Project, Konkola Copper Mines (KCM) in Zambia
    - Palabora Mining Company, South Africa
    - Dundee Precious Metals' Chelopech Mine, Bulgaria
    - Chuquicamata Copper Mine, Chile
    - Selebi-Phikwe, Botswana

- Eleven years (2006 – present) as the principal of Obsidian Consulting Services cc, an independent geological consulting company providing services to various mining and exploration companies around the world in the form of competent persons' reports, technical audits, planning and executing exploration programmes as well as advisory services. Apart from an audit for Volcan Minera's Cerro de Pasco mine in Peru, for 11 years I have been a contributor and signatory to the Glencore Alloys annual mineral resource and reserve statements for the chrome, vanadium and platinum operations in South Africa. Since 2014 I have been advising and serving as a Competent Person to a Bulgarian exploration company (non-disclosure agreements prevent me from releasing the clients name without their written consent) with extensive base metal projects located in Bulgaria. Since 2016 I have served as the Project Geologist for the Ngwenya Mining and Exploration (Pty) Ltd ferrous and base metals project in South Africa.

6. I have read the definition of a Qualified Person as set out in NI 43-101 as amended in 2011 and certify that by reason of my education, 26 years of relevant experience and professional affiliations, I fulfill the requirements to be a Qualified Person for the sections prepared by myself in the Report.

7. I have not visited the Haib Project site. The work submitted here has been prepared from data received electronically from Deep South Mining Company (Pty) Ltd and Haib Minerals (Pty) Ltd and I have not independently verified this data against originals.

8. As of the date of this certificate, I am not aware of any material fact or material change with respect to the subject matter of the Report, which is not reflected in the Report, the omission of which would make the report misleading.

9. In terms of section 1.5 of NI 43-101 "Standards of Disclosure for Mineral Properties" I am independent of the commissioning entities, being the Issuer, Deep-South Resources Inc, and its subsidiaries and associates Deep South Mining Company (Pty) Ltd and Haib Minerals (Pty) Ltd applying all of the standard tests of independence; Obsidian Consulting Services cc is also independent of the commissioning entities, their directors, senior management and advisors.

10. I have read NI 43-101 as amended on June 30, 2011 and confirm that this Technical Review Report has been prepared in compliance with the Standards and Guidelines as set out in that document.

D.S. RICHARDS *B.Sc. (Hons.) Pr. Sci. Nat. MGSSA*
Dated: 28th February 2018.

APPENDIX 1

Details of the Deep South Mining Company (Pty) Ltd

Exclusive Prospecting licence 3140

First Granted 22 April 2004

REPUBLIC OF NAMIBIA
MINISTRY OF MINES AND ENERGY

EXCLUSIVE PROSPECTING LICENCE
(Issued in terms of Section 70 of the Minerals (Prospecting and Mining) Act, 1992)

Exclusive Prospecting Licence No **3140** Office Reference No **14/2/1/4/2/3140**

Subject to the provisions of the Minerals (Prospecting and Mining) Act, 1992, this exclusive prospecting licence is hereby issued to

Full Name of Licence Holder **Deep-South Mining Company (Pty) Ltd.**

Identity or Passport No (natural person) **N/A**
Company Registration No (company) **2003/543**
Address (natural person) or Registered Address (company)
P.O. Box 22978
Windhoek, Namibia

Full Name of Accredited Agent (if applicable) **N/A**
Address of Accredited Agent (if applicable) **N/A**

for the period of **3 years** from (date of issue) **22 April 2004** to (date of expiry) **21 April 2007**

unless abandoned or cancelled on any prior date, or extended to such later date as may be endorsed on this licence in the event that this licence is renewed.

This exclusive prospecting licence is issued in respect of

Name of Mineral(s)/Group(s) of Minerals **Precious Metals and Base & Rare Metals**

over a certain portion of land situate in Region(s) **Karas**

Registration Division(s) **V** Magisterial District(s) **Karasburg**

as more fully depicted in the attached diagram No **EPL 3140** signed by the Commissioner

and is further subject to the terms and conditions contained in the notice of the Minister's intention to grant the

licence dated **20 April 2004** and agreed to in writing by the applicant on **22 April 2004**

as appended hereto.

Signed at WINDHOEK this 30th day of April 2004

MINISTRY OF MINES AND ENERGY
OFFICE OF THE MINISTER
2004 -04- 30
OFFICIAL STAMP
PRIVATE BAG 13297
WINDHOEK
OFFICIAL

....
MINISTER OF MINES AND ENERGY

11-0/0001

REPUBLIC OF NAMIBIA

MINISTRY OF MINES AND ENERGY

Tel (061) 284 8111
Fax (061) 238 643
E-mail: Post master@mme.gov.na
Web address: www.mme.gov.na

Private Bag 13297
WINDHOEK

Enquiries: Mr A Iilende

20 April 2004

Reference No.: 14/2/1/4/2/3140

Deep-South Mining Company (Pty) Ltd.
P. O. Box 22978
Windhoek
Namibia

NOTICE TO APPLICANT OF PREPAREDNESS TO GRANT APPLICATION FOR EXCLUSIVE PROSPECTING LICENCE 3140.

In terms of section 48(4) of the Minerals (Prospecting and Mining) Act, No. 33 of 1992, notice is hereby given that the Minister is prepared to grant your application, lodged on 01 December 2003 for an exclusive prospecting licence in respect of the Precious Metals and Base & Rare Metals Groups of Minerals over a certain area of land as shown in the attached diagram, subject to the terms and conditions contained in the attached schedule, which terms and conditions supplement the terms, conditions and provisions of the said Act.

Your attention is drawn to the provisions of section 48(5) of the said Act which require that within one month from the date of this notice, written acceptance of such terms and conditions must be received by the Commissioner, failing which the application will be deemed to have lapsed.

Kindly acknowledge your acceptance of such terms and conditions by-

(a) completing the section at the bottom of this notice;

(b) initialling each page of the schedule and the diagrams; and

All official correspondence must be addressed to the Permanent Secretary

(c) returning such signed and initialled documents to the Commissioner.

....
MINING COMMISSIONER

TO THE MINING COMMISSIONER
MINISTRY OF MINES AND ENERGY

I, John H. Akwenye (name of person) in my capacity as applicant/duly authorised ~~officer~~/approved accredited agent (please delete titles not relevant), hereby accept the supplementary terms and conditions referred to in this notice and contained in the attached schedule which are to be imposed on the grant of the application for exclusive prospecting licence herein referred to.

....
Signed

22-04-09
Date

Capacity... Director ....
(~~applicant~~/authorised officer of applicant if a company/~~approved accredited agent of~~ a ~~non-resident applicant who is a natural person/authorised officer of such accredited agent~~

SCHEDULE OF SUPPLEMENTARY TERMS AND CONDITIONS TO BE IMPOSED ON THE GRANT OF EXCLUSIVE PROSPECTING LICENCE NO. 3140 IN DEEP-SOUTH MINING COMPANY (PTY) LIMITED.

PART 1 - GENERAL

1. The exclusive prospecting licence shall endure for a period of three (3) years reckoned from the date of acceptance (hereinafter "the date of issue") of the terms and conditions referred to in this notice unless it is abandoned in terms of section 54 of the Minerals (Prospecting and Mining) Act, 1992, (hereinafter "the Act") or cancelled in terms of section 55 of the Act or on application made to the Minister in terms of section 72 of the Act, it is renewed by the Minister for any further period or periods.

2. In consideration of the rights hereby granted, the holder of the exclusive prospecting licence shall pay to the Commissioner for the benefit of the State Revenue Fund, such licence fee as may from time to time be prescribed in terms of section 123 of the Act, it being recorded that the annual licence fee prescribed in relation to the licence at the time of its issue shall be N$ 2 000 payable annually on or before each anniversary date of the date of issue of the licence.

3. In the event that the prescribed licence fee changes, such change shall become effective on the next anniversary date of the date of issue of the licence subsequent to such change.

4. The rights under the exclusive prospecting licence shall be limited in extent as stipulated in terms of paragraphs (d) to (g) of subsection 69(2) of the Act; provided that if during the currency of the exclusive prospecting licence, any claim area or area held under any other mineral licence existing on the date of issue of the exclusive prospecting licence which so limited such rights lapses, whether by abandonment, cancellation or expiry, such rights shall not extend to such claim or licence area.

5. The Commissioner may by notice in writing require the holder of the licence to beacon off the prospecting area in such a manner and within such a period, which shall not be less than one month, as may be specified in such notice at such holder's own cost.

6. The Minister may, in the interest of reasonable development of the prospecting operations, impose from time to time such additional conditions terms and conditions as he may deem fit.

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PART 2 - WORK PROGRAMME AND OBLIGATIONS

7. The holder of the exclusive prospecting licence shall-

7.1. commence with, and thereafter continue without undue interruption or delay, prospecting operations within one month of the date of issue of the licence in substantial conformity with the proposed work programme, schedule and budget which accompanied the original application for the licence and which served as motivation of the granting thereof;

7.2. where any material deviation of such work programme, schedule and budget is in the opinion of the holder of the licence, necessitated by the nature of the results of prospecting operations (but specifically excluding any circumstances of Vis Major provided for in terms of section 56 of the Act), apply in writing to the Minister for approval of the revision of such work programme, schedule and budget in terms of section 75 of the Act;

7.3. execute such additional work programme and expend such additional expenditure within a specified period of time as may be imposed by the Minister from time to time;

7.4. the holder of the exclusive prospecting licence shall be obliged to secure a Joint Venture partner (who has the technical and financial resources) within one year of the date of issue of this licence; and

7.5. the holder of the exclusive prospecting licence shall give a presentation to the Ministry of Mines and Energy regarding the progress made on exploration, within one year of the date of issue of the licence.

PART 3 - ENVIRONMENT

8. The holder of the exclusive prospecting licence shall observe any requirements, limitations or prohibitions on his or her prospecting operations as may, in the interest of environmental protection be imposed by the Minister from time to time.

9. The holder of the exclusive prospecting licence shall enter into an Environmental Contract with the Ministry of Environment and Tourism and that of Mines and Energy within one (1) month of the date of issue of the licence

....

MINING COMMISSIONER

DATE

MINISTRY OF MINES AND ENERGY
MINING COMMISSIONER
20 APR 2004
Private Bag 13297
9000 WINDHOEK
OFFICIAL

08 SEP '04 (WED) 14:44 COMMUNICATION No. 32 PAGE 7

APPENDIX 2

Details of the Deep South Mining Company (Pty) Ltd

& Successor in Title, Haib Minerals (Pty) Ltd

Exclusive Prospecting licence 3140

After Renewal on 22 April 2017

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

With the approval of the Minister of Mines and Energy, this mineral licence has been renewed for a period of Two (2) years from 22 April 2007 to 21 April 2009

15.09.2008
Date
Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

ith the approval of the Minister of Mines and Energy, this neral licence has been renewed for a period of TWO (2) years from 2 April 2009 to 21 April 2011

/04/2009
te
Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

With the approval of the Minister of Mines and Energy, this mineral licence has been renewed for a period of Two (2) years from 22 April 2011 to 21 April 2013

-04-2011
ate
PP Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (ALIENATION)

With the approval of the Minister of Mines and Energy, this licence/~~an interest in this licence~~ has been alienated by transfer/~~grant/cession/assignment~~ to, ~~or the joinder as a joint holder of~~ Haib Minerals (Pty) Ltd

with effect from 12 April 2013

22/04/2013
Date
Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

With the approval of the Minister of Mines and Energy, this mineral licence has been renewed for a period of Two (2) years from 22 April 2013 to 21 April 2015

27/05/2013
Date
PP Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

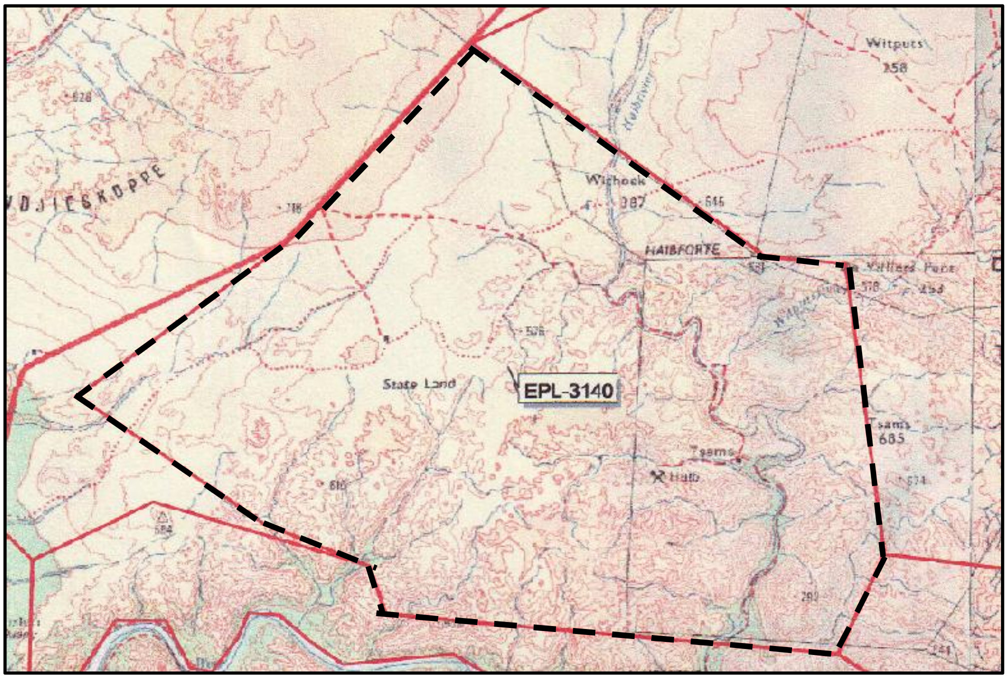
With this approval of the Minister of Mines and Energy, this mineral licence has been renewed for a period of Two (2) years from 22 April 2015 to 21 April 2017

01/04/2015
Date
PP Commissioner

MINISTRY OF MINES AND ENERGY
ENDORSEMENT (RENEWAL)

With this approval of the Minister of Mines and Energy, this mineral licence has been renewed for a period of Two (2) years from 22 April 2017 to 21 April 2019

11/09/2017
Date
Commissioner



Licence	Nr	Lat	Long
EPL-3140	1	-28.72530758	17.78740212
EPL-3140	2	-28.71183732	17.74992010
EPL-3140	3	-28.67100381	17.68364770
EPL-3140	4	-28.62124926	17.75614951
EPL-3140	5	-28.55535382	17.82202069
EPL-3140	6	-28.62443708	17.92656044
EPL-3140	7	-28.62670927	17.95497900
EPL-3140	8	-28.69475300	17.96558957
EPL-3140	9	-28.72089474	17.96970697
EPL-3140	10	-28.75324360	17.95375930
EPL-3140	11	-28.74086885	17.79335440

The current map of EPL 3140 and a table of the coordinates.
The current area is 36,502.4ha.

APPENDIX 3

Legal Opinion Letter on the Validity of

Exclusive Prospecting licence 3140

As at 12th December 2017

H. D. BOSSAU & CO

Legal Practitioners / Notaries

49 Feld Street
Windhoek
Republic of Namibia

Telephone (+264)(061) 370 850
Facsimile (+264)(061) 370 855
P.O. Box 1975,
Email NicoDP@bossau.com

YOUR REF:

OUR REF:
DEE1/0005

12 December 2017

To:

Peter Walker
P & E Walker Consultancy CC,
41 Dennekamp,
Main Road, Kenilworth,
7708
South Africa

Dear Sirs,

RE: Namibia: Legal Opinion on the status of certain Exclusive Prospecting Licence Number 3140 issued by the Ministry of Mines and Energy of the Republic of Namibia to Haib Minerals (Proprietary) Limited

---

1. We have been requested by Haib Minerals (Proprietary) Limited ("**Haib**" or "**the Company**") to advise on the under mentioned matters.

2. We render this opinion, which is given subject to the following conditions and qualifications –

   2.1 This opinion is confined to matters of Namibian law and no opinion is expressed as to the laws of any other jurisdiction. We do not purport to be experts on and do not purport to be generally familiar with or qualified to express legal opinions based on any law other than the laws of the Republic of Namibia ("**Namibia**"), and accordingly, express no legal opinion herein based upon any other laws.

2.2 Statements made herein relating to the Company, the Licence and prospecting and/or mineral rights, and other matters dealt with in this opinion are based upon information submitted to us and information obtained from the records of the Ministry of Mines and Energy ("**the Ministry**"). We cannot ourselves vouch for the accuracy of the information so obtained.

2.3 In giving this opinion, we have assumed:

(a) the genuineness of all signatures on documents submitted to us as originals or copies of originals;

(b) the authenticity and completeness of all documents submitted to us as copies including the actual licence documentation in the form of Exclusive Prospecting Licence Number 3140 ("**EPL 3140**" or "**the Licence**"), and that such copies are, in fact, true copies of documents in existence and that the originals of such documents were properly executed and the authenticity and completeness of such originals;

(c) the accuracy of the official records maintained by the Ministry tendered to us for investigation and perusal. We point out that the Ministry only facilitated access to the following documents:

i. Licence contacts reflecting an active Exclusive Prospecting Licence registered under Licence Code 3140;

ii. Licence contacts reflecting an application for an Mineral Deposit Retention Licence under Licence Code 3140;

iii. Application for an Exclusive Prospecting Licence dated 20 April 2017 ("**EPL Application**");

iv. Application for a Mineral Deposit Retention Licence dated 20 January 2017 ("**MDRL Application**");

v. Licence Details of the Exclusive Prospecting Licence (Exploration) EPL – 3140 dated 21 November 2017; and

vi. Licence Details of Exclusive Prospecting Licence 5327 ("**EPL 5327**")

and not to any related documentation, such as other or further applications, reports and/or correspondence;

(d) that the documents dated on or earlier than the date hereof and upon which we have expressed reliance remain accurate;

(e) that the documents examined are the only documents pertaining to title to the Licence;

(f) that the persons purporting to execute the documents examined in the course of the title examinations are, in fact, the same persons named therein and, when executed by a corporation or regulatory authority or official, that the persons so executing were duly authorised as signing officers.

2.4 In giving this opinion:

(a) no investigation has been made of the original applications for filing or renewal (except for the EPL Application in respect to the renewal of the Licence and the MDRL Application), the location of the boundaries of the Licence or the existence of any interest in the Licence other than what was noted on the Licence submitted to us and the Licence Details as made available to us and maintained in the office of the Ministry at Windhoek, on 17 November 2017;

(b) no investigation has been undertaken as regards compliance by the Company with the terms and conditions attaching to the Licence, or compliance with the provisions of the Minerals (Prospecting and Mining) Act, Act 33 of 1992 ("**the Act**");

(c) we have not for the purpose of this opinion examined any contracts, instruments or other documents entered into by or affecting the Company, or the rights comprised in the Licence or any corporate records of the Company and have not made any other enquiries concerning the Company, the Licence or the mineral rights in terms thereof.

2.5 This opinion is given to you for use in connection with the verification of the mineral licence rights of the Company in Namibia. It may not be relied upon by any other person or used for any other purpose and neither its contents nor its existence may be disclosed without our prior written consent.

3. In regard to the matters requested by you, we are of the opinion that:-

3.1 The Licence

(a) As stated above, we inspected the Licence submitted to us during the course of November 2017.

(b) We also attach the following documents marked Schedules I - III, obtained from the Ministry on 17 November 2017, which reflect all of the particulars of the Licence:

I. A printout from the electronic data base in respect to the Licence contacts registered under Licence Code 3140;

II. A copy of the Application for an Exclusive Prospecting Licence dated 20 April 2017;

III. A printout from the electronic data base in respect to the Licence Details of the Exclusive Prospecting Licence (Exploration) EPL – 3140 (3140) dated 21 November 2017.

(c) Furthermore we noticed that there is another Exploration Prospecting Licence registered in the same prospecting area under number 5327, and requested a printout from the electronic data base in respect to the License Details of the Exclusive Prospecting Licence (Exploration) EPL – 5327 (5327) dated 30 November 2017, marked schedule IV.

(d) In addition to the details appearing on Schedule I - III we set out below some particulars gleaned from the Licence inspected.

(e) Exclusive Prospecting Licence 3140 –

(i) bears the official reference number 14/2/1/4/2/3140;

(ii) reflects the Company as licence holder;

(iii) was originally dated 20 April 2004 and initially issued with effect from 22 April 2004 to 21 April 2007 and was endorsed to reflect the following renewals –

A. with effect from 22 April 2007 until 21 April 2009,

B. with effect from 22 April 2009 until 21 April 2011;

C. with effect from 22 April 2011 until 21 April 2013;

D. with effect from 22 April 2013 until 21 April 2015;

E. with effect from 22 April 2015 until 21 April 2017;

F. with effect from 22 April 2017 until 21 April 2019;

(iv) is granted in respect of base and rare and precious metals. However we wish to point out that the Licence excludes Industrial Minerals which were applied for in the EPL Application;

(v) is granted in respect of a certain portion of land situated in the Karas Region, Registration Division V, in the Magisterial District of Karasburg, as depicted in the diagram No. EPL 3140 attached to the Licence;

(vi) diagram No. EPL 3140 attached to the most recent renewal reflects the Licence area as being 36589.1879 hectares in extent;

(vii) is subject to the supplementary terms and conditions contained in the most recent Notice of the Minister of Mines and Energy's Preparedness to Renew the Licence dated 28 August 2017 in respect of a renewal application lodged on 20 April 2017, a copy whereof is appended to the Licence, entitled as more fully described in paragraph (viii) immediately below;

(viii) is subject to the terms and conditions set out in the schedule entitled "SCHEDULE OF SUPPLEMENTARY TERMS AND CONDITIONS TO BE IMPOSED ON THE GRANT OF THE RENEWAL OF EXCLUSIVE PROSPECTING LICENCE NO. 3140 In Favour Of Haib Minerals (Pty) Ltd" ("**the Terms and Conditions**");

(ix) reflects that the Minister's Notice of Preparedness to Renew mentioned in paragraph (vii) was accepted by the Company on 31 August 2017;

(x) does not reflect any encumbrances, liens or charges of any kind. However, there is no way of ascertaining whether the Licence has been encumbered, or is subject to any lien or charge, as there is no mechanism for this to be recorded in any public office, since the legislation does not provide for this.

(d) The Company is the registered holder of the Licence, which has on the face thereof been validly granted in accordance with the applicable laws and which is in effect and valid in accordance with its terms.

3.2 While we urge you to have regard to the full terms and conditions imposed in respect of the Licence reported on, we would briefly like to comment on the content thereof, as follows:

(a) The first part of the Terms and Conditions deals with the term for which the Licence is granted, being a period of two (2) years, and prescribes the annual fee that is payable by the Licence holder, namely NAD4,000.00 (four thousand Namibia Dollar). It is also provided that in the event of an increase in fees during the term of the Licence, the increased fee is payable with effect from the next anniversary date of the renewal of the Licence, subsequent to such change. The terms and conditions then further draw specific attention to the provisions of Section 69(2)(d) to (g) of the Act. This in essence means that the Licence does not extend in respect of minerals or groups of minerals for which it is not granted, and also specifically excludes any pre-existing mineral rights belonging to third parties that may be in existence in the Licence area. It is also provided that where there are pre-existing third party rights which lapse during the term of the Licence within the Licence area, these shall be included as part of the Licence. It is also provided that the Minister may in the interests of the reasonable development of the prospecting operations impose additional terms and conditions on the Licence holder.

(b) The second part of the terms and conditions provides that the licence holder shall commence and continue with the proposed work programme, schedule and budget of expenditure to be incurred which accompanied the application for the Licence (or, if applicable, the application for renewal thereof) and which served as the motivation for the granting thereof. Any material deviation from such work programme, schedule or budget requires notification to the Minister, and Ministerial approval for any proposed revision in terms of section 75 of the Act. There is also a condition to the effect that the Minister may impose on the licence holder the obligation to undertake an additional work programme or additional expenditure within a specified period as may be prescribed from time to time.

(c) There is another condition to the effect that all funds raised anywhere and exclusively in respect of this Licence shall be expended on the Licence and all/any activities relating thereto, to the extent that such funds are to be expended directly in Namibia, the Licence holder shall ensure such funds are remitted to a reputable financial institution in Namibia.

(d) The Licence holder is furthermore required to make an oral presentation to the Ministry after the first year of the Licence tenure.

(e) The third part stipulates that the Licence holder must observe all requirements, limitations or prohibitions imposed on the prospecting operations as may in the interests of environmental protection be imposed by the Minister from time to time. It is furthermore provided that the Licence holder shall adhere to the environmental contract with the Ministry of Environment and Tourism and the Ministry of Mines and Energy already concluded in respect of the Licence.

(f) In terms of the Environmental Management Act, 2007, additional requirements as regards compliance with this Act should now also be complied with by the Licence holder. We have not been instructed to investigate such compliance and have accordingly not done so.

(g) The fourth part prescribes additional conditions which must be met within 60 days after the acceptance of the Notice of Preparedness, by submitting a detailed description on the structure and composition for the Company, together with a proposal (the "**Proposal**"), addressing the Government's objectives of poverty eradication, to the Minister. We wish to point out that the additional conditions had to be complied with on or before 30 October 2017. We are not informed as to whether this was complied with or not.

(h) The conditions in respect to the structure and composition of the Company set out that the management structure of the Company shall be represented by a minimum of 20% historical disadvantaged Namibians and a percentage of at least 5% of the shareholding of the Company shall be held by Namibian persons or a company wholly owned by Namibians.

(i) The Proposal should further address the Government's objectives of poverty eradication by providing an opportunity for Namibian participation, as well as setting out a strategy to benefit the Namibian youth and women particularly from the disadvantaged groups and the poorest of the poor. We have not been instructed to investigate such compliance and have accordingly not done so.

(j) The Minister may subsequently propose amendments within 30 days, after receiving the proposal, which would enable the Company to support the Government's objectives for broad based empowerment and poverty eradication. Where the project is economically significant, but the proposed structure of the Company does not meet the Government's objectives, the Minister shall have the right to propose amendments in writing to the Company, to meet these objectives of the Government.

(k) If the Company is dissatisfied with the counter-proposal of the Minister it had to make written representations to the Minister within 30 days from the date of the Minister's counter proposal and after consideration, the Minister should have notified the Company of the final terms and conditions on which the Minister is prepared to grant the Exclusive Prospecting Licence.

(l) As indicated above, we are unable to confirm that the Company has indeed complied with all these Terms and Conditions or any other terms and conditions provided for in the Act. However, as appears from paragraph 3.5 below, compliance with the applicable conditions is one of the aspects that the Minister must be reasonably satisfied with prior to granting the renewal of a licence. In the absence of compliance it is unlikely that the Licence would have been renewed on the several occasions that it has been renewed, including for its current tenure.

3.3 As regards renewals, we point out that renewal applications shall be made not later than ninety (90) days before the date on which a Licence expires, or such later date, but not later than the expiry date of the Licence, as the Minister may on good cause shown allow. Where an application for renewal is made late, it is therefore necessary to show cause and request condonation for the late application. It appears that the latest renewal was only applied for on 20 April 2017, that is, one day before the then current term expired. However, prior to this, on 20 January 2017 the Company had submitted the MDRL Application and as such there appears to have been a renewal application as a cautionary measure to preserve the existing rights. The lateness of the renewal application in our view therefore should not pose a problem.

It is furthermore provided that in the case of a first application for the renewal of a Licence it shall not be made for an area greater in extent than seventy five percent (75%) of the prospecting area comprising the initial area of grant, and, in the case of further renewal applications, in respect of any land greater in extent than fifty percent (50%) of the prospecting area existing at the date of such application, unless the approval of the Minister has been obtained to retain a larger area. The Minister may on renewal approve a larger area in the interests of the development of the mineral resources of Namibia and on good cause shown by the holder of the licence in question. No reduction was applied for with the renewal, but again in our view this should be linked to the MDRL Application and accordingly acceptable.

The Minister shall also not grant an application for renewal unless the Minister is on reasonable grounds satisfied with the manner in which the programme of prospecting operations has been carried out, and the expenditure expended in respect of such operations.

While a renewal application is being considered by the Minister the exclusive prospecting licence which is the subject of such renewal application remains valid, even where the initial term has expired, until such time as the outcome of the renewal application has been determined.

We also wish to address the provisions of the Act dealing with renewals, and more particularly the acceptance by a licence holder of a Notice of Preparedness to Grant a Renewal issued by the Ministry. The provisions of section 48(5) provide that within one month as from the date of a Notice of Preparedness, or such further period as the Minister may on good cause shown allow in writing, the applicant must agree in writing to accept the terms and conditions attaching thereto or such other terms and conditions as may be agreed upon, failing which the application in question shall lapse on the expiration of such period. As per paragraph 3.1(c)(ix) this was complied with if regard is had to the most recent acceptance of the Notice of Preparedness to Grant a Renewal by the Company within three days of the Ministry's notice.

Any exercise of discretion by the Minister must be undertaken in accordance with the general principles of administrative justice, and the Minister is expected to act fairly and reasonably in doing so. This is relevant to the consideration, among others, of any renewal applications in relation to the Licence.

All administrative acts are presumed to have been lawfully done or performed until proof to contrary has been adduced. It is technically possible for a third party to challenge licence rights on the basis that the third party's rights have been infringed or some administrative process was not properly followed, or a discretion was not properly exercised. However, until successfully challenged by someone with the legal standing to do so (that is, some right or connection to the licence and the Ministerial decision in question) and set aside by a court of law, a licence is at least *prima facie* valid. An application to have the licence set aside must also be brought within a reasonable time. The rule of practice has developed that this means that an application must be launched within three (3) to six (6) months from the date of grant of a licence. The purpose of this time limitation is to arrive at finality and create certainty. In any review application challenging the grant of a licence the Ministry would be obliged to make available a complete record of the decision making process, comprising all documentation in relation to the decision that was taken. By virtue of this discovery process it is therefore possible for an applicant to extract information, and even an application brought on flimsy grounds could as a result be substantially amplified and raise new review grounds where there were shortcomings that appear from the record.

3.4 A licence holder must also enter into a surface rights usage agreement in respect of the land over which the licence rights, or some of them, extend; provided that the land in question is privately owned. In terms of section 52 of the Act a licence holder must have an agreement in writing with the landowner before being in a position to exercise licence rights. Where no such agreement is in place the licence holder may be granted an ancillary right as provided in section 110(4) of the Act. An ancillary right may be applied for in terms of section 109. The Ancillary Rights Commission constituted by the Act has sittings at times determined by it, and usually is convened on a quarterly or half yearly basis. An application for a grant of

ancillary rights is therefore likely to be coupled to delays. In instances where licence rights extend over State Land no surface right agreement is required.

3.5 We would also like to point out that in terms of section 47(3) of the Act the provisions of section 39(6), (7) and (8) of the Act apply in the case of a transfer, cession, assignment or grant of an interest in a mineral licence. Accordingly, all rights, liabilities and obligations which vested in the licence holder immediately before such transfer, cession, assignment or grant, shall vest in the new licence holder from the date of the transaction. This includes all pre-existing environmental damage.

3.6 Kindly be advised that Ministerial approval is required where a transfer, cession, assignment or grant of an interest in a mineral licence is contemplated. Where a transfer of shares in the licence holder is contemplated, only notification of the change in shareholders is required.

3.7 In this context it should be noted that the Act also affords the Minister the power to cancel a mineral licence in certain circumstances. In respect of corporate entities the grounds are limited to a failure to comply with the terms and conditions of the mineral licence or of the provisions of the Act, and where a company is wound up in terms of the provisions of the Companies Act, 2004, other than for the purposes of amalgamation or reconstruction undertaken with the prior approval of the Minister. A licence may not, however, be cancelled unless the Minister has first given written notice to the licence holder informing the licence holder of the intention to cancel the mineral licence and setting out particulars of the failure complained of as well as calling on the holder to make representations. The Minister is also obliged to consider any steps taken by the holder to remedy the failure in question or to prevent any such failure from being repeated during the currency of the mineral licence and any other matter submitted by way of representations. Non-compliance by the licence holder due to *vis major* is not, in terms of the Act, regarded as a failure to comply with terms and conditions or provisions, provided that proper notice must be given of such events.

3.8 Based on the records inspected by us, we are not aware of any breaches being noted thereon in respect of the Terms and Conditions attaching to the

Licence by virtue of the Licence Terms and Conditions themselves or in terms of the Act.

4. In regard to MDRL Application registered under Licence Code 3140:

4.1 The MDRL Application

a) As stated above, we inspected the MDRL Application at the office of the Mining Commissioner, at the Ministry, on 17 November 2017. We also attach, marked Schedule V, a copy of the application obtained from the Ministry on 17 November 2017 herewith.

b) In addition to the details appearing on Schedule V we set out below some particulars gleaned from the MDRL Application inspected.

c) The MDRL Application--

(i) bears the official reference number 14/2/1/4/2/3140;

(ii) reflects the Company as Applicant;

(iii) was filed on 20 January 2017;

(iv) is in respect of precious metals, base and rare metals, dimension stone, industrial minerals, precious stones and semi-precious stones;

(v) is in respect of a certain portion of land situated in the Karas Region, Registration Division V, in the Magisterial District of Noordoewer, as depicted in the diagram attached to the MDRL Application.

4.2 Kindly take note that the MDRL Application is still pending the approval of the Minister of Mines and Energy.

4.3 Although we inspected the MDRL Application we cannot confirm the accuracy or completeness of the information provided therein.

5. In regard to the prospecting area of the Licence and the MDRL Application, we also wish to draw your attention to EPL 5327 which we inspected at the Ministry of Mines and Energy on 30 November 2017.

5.1 In addition to the details appearing on Schedule IV we set out below some particulars gleaned from the Licence Details of EPL 5327 as inspected.

a) Exclusive Prospecting Licence 5327 –

i. reflects the licence holder to be Namibia Hau Yan Resources Explo & Devel (Pty) Ltd;

ii. it appears from the Licence Details that an application was initially filed on 17 May 2013. Subsequently the licence holder filed an application for a renewal on 14 October 2016;

iii. the status of the renewal is still pending and the application had not been approved on the date we inspected the Ministry's available records;

iv. the initial EPL 5327 was granted in respect of industrial minerals and precious stones.

5.2 We wish to point out that the Act prescribes that the Minister may grant to any person in respect of the prospecting area to which an exclusive licence relates an exclusive prospecting licence, mining licence or mineral deposit retention licence in respect of any mineral or group of minerals other than the mineral or group of minerals to which such first-mentioned exclusive prospecting licence relates.

It appears from the licence details of EPL 5327 that Namibia Hau Yan Resources Explo & Devel (Pty) Ltd, holds an exclusive prospecting licence for industrial minerals since 29 October 2013. This presumably explains why the Ministry denied the Company's application for industrial minerals to be included in the Licence area as per the EPL Application.

Furthermore we noticed from the MDRL Application that the Company applied to include precious stones as a commodity. Kindly take note that Namibia Hau Yan Resources Explo & Devel (Pty) Ltd currently holds an exclusive prospecting licence for precious stones and industrial mineral over the same area.

The Company will therefore not be able to apply for a any licence in the prospecting area covered by EPL 5327 in respect to industrial minerals or precious stones, unless the Minister refuses the renewal application of Namibia Hau Yan Resources Explo & Devel (Pty) Ltd which is pending.

5.3 We also noticed further that the Company applied for the inclusion of dimension stone in the MDRL Application.

The Act prescribes that no person shall apply for a mineral deposit retention licence, unless such person is the holder of an exclusive prospecting licence or mining claim in relation to the area of land and the mineral or group of minerals to which his or her application relates.

According to Section F of the MDRL Application, the Company holds no other mineral licences including claims, except for EPL 3140, under which Licence number the MDRL Application is brought.

It is therefore our view that the Ministry may only approve the Company's MDRL Application in respect of base and rare and precious metals, provided the Minister is satisfied with all the prescribed requirements in the Act in respect to the said Application, are complied with.

Notwithstanding the above, the Company may apply to the Ministry to amend its Licence to include dimension stone, and should thereafter apply for the amendment of the mineral deposit retention licence (once approved by the Minister), to include dimension stone.

Kindly take note that an exclusive prospecting licence should be amended every time the licence holder wishes to add an additional mineral or group of minerals to its licence rights.

We trust that this opinion is of assistance to you and remain available should you have any further questions herein.

Yours faithfully,

H.D. Bossau & Co.
Per:

N.C. du Plooy