This report has been prepared by Snowden Mining Industry Consultants (‘Snowden’) on behalf of North River Resources Plc.

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Prepared By
J Elkington ........................................................................................
M.Sc. (Min Econ), MAusIMM, AWASM
General Manager - UK

J Priest..............................................................................................
M.Eng. B. Eng. (Mining); C.Eng; MIMMM, PMP, SCPM
Principal Consultant - Mining

P J Petit
Senior Consultant – Corporate Advisory

R Moserne ........................................................................................
B.Sc. Eng. (Mining); GDE; SAIMM; ECSA
Senior Consultant - Mining

M Seymour .......................................................................................
B.Sc. (Geology), MSc (Mine Geomechanics), MAusIMM, RPEQ
Principal Consultant

Reviewed By
J Elkington ........................................................................................
M.Sc. (Min Econ), MAusIMM, AWASM
General Manager - UK

J Priest..............................................................................................
M.Eng. B. Eng. (Mining); C.Eng; MIMMM, PMP, SCPM
Principal Consultant - Mining

W F McKechnie .................................................................................
B.Sc. (Hons), MSAIMM, FGSSA, Pr.Sci.Nat
Senior Principal Consultant

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Appendices

Appendix A   NRR In-Situ Namib Lead Zinc Project Resource Estimate
Appendix B   Ventilation Studies
Appendix C   Risk Assessment Report
Appendix D   Environmental Review
1 Executive summary

1.1 Introduction

On behalf of North River Resources Plc. (NRR), Snowden Mining Industry Consultants (Snowden) undertook the preparation of a Mine Development Plan (MDP) for the production of lead, zinc and silver from the Namib Lead Zinc Project (the Project) area in Namibia. This study envisaged reopening, mining and processing the existing Namib underground mine, North and N20 Lodes, construction of a crushing, grinding and flotation process circuit and the products being transported to Walvis Bay for shipment to market.

The directors of NRR requested that a mining schedule of 20 thousand tonnes per month (ktpm) be prepared using Mineral Resources in the Indicated category to provide a Mining Inventory sufficient to sustain a plant throughput of 20 ktpm. **This mining inventory is not a Reserve as it does not conform to the JORC 2012 guidelines.**

It is considered by the directors of NRR that future exploration success will also contribute to the Project’s financial future. This assumed exploration success is based on the brownfields geological setting and the potential for upgrading of a substantial portion of the existing Inferred Mineral Resources to an Indicated classification. Also further success may be anticipated from drilling intercepts on extensions of the presently defined mineralisation at depth.

The contributors to this study and their tasks are presented in Table 1.1.

<table>
<thead>
<tr>
<th>Contributor</th>
<th>Section</th>
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<tr>
<td>Snowden</td>
<td>Resources, mining, mining Capex, Opex and mining infrastructure, metallurgical and process review, cashflow model, financial evaluation, risk analysis and assembling the report</td>
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<td>Tenova Bateman</td>
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<td>Celtis Geotechnical</td>
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<td>NRR</td>
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</table>

1.2 Headline financial statistics

Snowden prepared a cashflow model to provide an indication of the overall economics of the project given the supplied inputs. This cash-flow model provided the overall project financial statistics after taxation and are presented in Table 1.2.
### Table 1.2 Financial statistics

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<th>Parameter</th>
<th>Units</th>
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<td>USDM</td>
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<td>Revenue</td>
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<td>Cash outflow</td>
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<td>Operating cash-flow</td>
<td>USDM</td>
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<td>IRR</td>
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### 1.3 Project description and scope

NRR acquired the Project in 2009. The Project comprises three main components:

- existing surface tailings dumps
- existing underground mine workings and mineralised down dip extensions (South)
- North and N20 lodes that represent unmined extensions (North).

NRR initiated the refurbishment of access ways and services in the underground workings. Subsequent to the re-establishment of services, and confirmation of structural integrity, the underground workings and all accessible voids were surveyed and survey control was re-established throughout the mine.

To increase confidence in the Resources, NRR commenced an infill drilling programme. Following encouraging results, NRR decided to facilitate further exploration to increase Resources below the existing mine and also the North lode extensions. An in-house mineralisation map was completed and a channel sampling program in the existing mine was also undertaken.

The initial Project plan envisaged a mining and processing schedule, a capital budget estimate and operating costs to construct a processing plant with the aim of processing tailings at 300 kilo tonnes annually (ktpa), until extracted material from the underground mine became available. A metallurgical test work, drilling and sampling programme was undertaken on the tailings dump but initial results indicate that the tailings resource has less economic potential than originally understood. The directors of NRR advised that further testwork is being completed to re-assess the economics of the tailings; however, for the purposes of this study Snowden were instructed to exclude the tailings.

This study includes a detailed mine schedule matched to the planned processing plant with a capital and operating costs budget estimate for mining, surface and underground supporting infrastructure and services to sustain the operation. The production schedule is based upon an approximately 1 million tonnes (Mt) mining inventory of the current Inferred and Indicated Mineral Resources.
1.4 Location and climate

1.4.1 Location

The Project is located approximately 28 kilometres (km) north-northeast of Swakopmund and 8 km north of the main Swakopmund-Windhoek highway (Route B2) (Figure 1.1). The site is accessed via a well-marked gravel road which passes over the Namib rail line from Swakopmund to Windhoek as shown in Figure 1.1.

The general topography is characterised by generally scattered, low hills and inselbergs.

Figure 1.1 Mine location

Source: http://www.nationsonline.org/oneworld/map/namibia-political-map.htm)
1.4.2 Climate

Climate in the area is generally described as semi-arid desert being sunny and windy, with fog conditions stretching inland at times.

Average annual daytime air temperature is 19.6 degrees Celsius, with a minimum of approximately 16 degrees Celsius experienced in August, and a maximum of approximately 21 degrees Celsius. The temperature is kept moderate by the south westerly to south-south westerly wind, with an average wind probability of 3.1 on the Beaufort scale (Figure 1.2). The average wind speed is 5 knots or 9.26 km per hour.

It is to be noted that the default units for wind and temperature, are knots and degrees Celsius, respectively. The dominant wind direction is shown by the arrow in Figure 1.2.

Figure 1.2  Swakopmund wind and temperature statistics

![Wind and Temperature Statistics](http://www.windfinder.com)

Annual average rainfall measured over the last seven years averaged 38.4 millimetres (mm). The wettest month is February, averaging 6.3 mm. The driest month is December, averaging 0.4 mm (Figure 1.3).
Figure 1.3  Swakopmund rainfall statistics

<table>
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<td>3.0</td>
<td>-</td>
<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
<tr>
<td>2018</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>4.0</td>
<td>3.0</td>
<td>3.0</td>
<td>-</td>
<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
<tr>
<td>2019</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>4.0</td>
<td>3.0</td>
<td>3.0</td>
<td>-</td>
<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
<tr>
<td>2020</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>4.0</td>
<td>3.0</td>
<td>3.0</td>
<td>-</td>
<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
<tr>
<td>2021</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>4.0</td>
<td>3.0</td>
<td>3.0</td>
<td>-</td>
<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
<tr>
<td>Avg</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>4.0</td>
<td>3.0</td>
<td>3.0</td>
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<td>2.6</td>
<td>2.6</td>
<td>4.0</td>
<td>3.0</td>
<td>2.6</td>
<td>64.2</td>
</tr>
</tbody>
</table>

Source: http://www.namsearch.com
1.5 **Resource Estimation study (2013)**

(See Appendix A for the full Resource Estimation report)

NRR engaged Snowden to undertake a study on the Namib Lead Zinc Project (the Project), on its behalf. Dr Simon Dominy is an Executive Consultant in Applied Geosciences at Snowden and is the appointed Competent Person (CP) for this Resource.

NRR is a sole holder of EPL 2902 issued for the Project, located 28 kilometres (km) north-northeast of Swakopmund, Namibia. The project area hosts lead-zinc mineralisation which has been mined historically through underground workings between 1965 and 1992, from the Namib lead mine.

The purpose of this report is to serve as supporting documentation to the 2013 Mineral Resource Estimate for the Project in-situ mineralisation, reported in accordance with the JORC 2012 Code. The Resource was estimated under the guidance of the CP as defined by the JORC 2012 Code.

The Project is composed of separate bodies with an overall northwest trend and steeply southwest dipping lead (Pb) and zinc (Zn) mineralisation associated with regional sedimentation and metamorphism. Geological, structural and mineralisation models were created based on historical and current 2D and 3D datasets. Separate mineralised ore lodes were modelled using a 1.00% PbZn (Pb% + Zn%) cut-off. A 1 metre (m) composite field was used in a geostatistical study (Variography and Quantitative Kriging Neighbourhood Analysis – QKNA) that enabled Ordinary Kriging (OK), controlled by dynamic anisotropy, to be used as the interpolation method. The results of the variography and the QKNA were utilised to determine the most appropriate search parameters and sample numbers.

The parent block size used for the estimation was 4 mE by 4 mN by 4 mRL, with sub-celling down to 1 mE by 1 mN by 1 mRL. The interpolated block model was validated through visual checks, a comparison of the mean composite and block grades and through the generation of section validation slices.

Previously a Mineral Resource, reported in accordance with the JORC code, was prepared in 2012 by CSA Global Resource Industry Consultants (CSA), for the in-situ Project, which yielded the following results for Pb, Zn, Ag (silver) and In (Indium):

<table>
<thead>
<tr>
<th>Table 1.3</th>
<th>In-situ Project Mineral Resource Estimate by CSA (2012) - Undepleted</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Class</strong></td>
<td><strong>Area</strong></td>
</tr>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
</tr>
</tbody>
</table>

*Tonnages have been rounded to the nearest 1,000 t to reflect an estimate*
The 2013 Project Resource model is rebuilt from the 2012 Project Resource model based on the re-interpretation of the mineralisation, informed by historic drillhole data giving confidence in continuity of mineralisation in the Northern Extension and the incorporation of new drilling and channel sampling results in the South Mine.

Snowden estimated three elements, namely Zn, Pb and Ag using CAE Datamine Studio 3™. The results are as follows:

### Table 1.4  In-situ Project Mineral Resource Estimate by Snowden (2013) - Undepleted

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>*Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
<td>529,000</td>
<td>3.45</td>
<td>2.8</td>
<td>5.4</td>
<td>48.2</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>139,000</td>
<td>3.45</td>
<td>2.0</td>
<td>4.3</td>
<td>42.4</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
<td>253,000</td>
<td>3.45</td>
<td>1.8</td>
<td>7.2</td>
<td>39.0</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>7,000</td>
<td>3.45</td>
<td>2.2</td>
<td>3.5</td>
<td>53.4</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>928,000</td>
<td>3.45</td>
<td>2.4</td>
<td>5.7</td>
<td>44.9</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1,000 t to reflect an estimate

The block grade estimates compare well with the composites, as well as between the 2012 and 2013 models. The major increase in tonnage is due to the increase in the modelled Resource volumes in the 2013 rebuild. The Northern Extension ore lodes were re-interpreted, incorporating structural and geological data, as well as Down Hole Transient Electromagnetic (DHTEM) plates as modelled by Southern Geosciences Consultants (SGC). Additional historical drillholes were used for the interpretation as indication to the continuity of ore lodes. The South Mine area showed an increase in the modelled volume due to additional drillhole and channel sampling data.

### Table 1.5  Comparison between the modelled Resource volumes for 2012 and 2013

<table>
<thead>
<tr>
<th>Model</th>
<th>Area</th>
<th>*Volume</th>
<th>Density (t/m³)</th>
<th>*Tonnes</th>
</tr>
</thead>
<tbody>
<tr>
<td>2012</td>
<td>Northern Extension</td>
<td>175,000</td>
<td>3.45</td>
<td>603,000</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>34,000</td>
<td>3.45</td>
<td>116,000</td>
</tr>
<tr>
<td>2013</td>
<td>Northern Extension</td>
<td>247,000</td>
<td>3.45</td>
<td>853,000</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>43,000</td>
<td>3.45</td>
<td>149,000</td>
</tr>
</tbody>
</table>

*Volumes and Tonnages have been rounded to the nearest 1000

Grade-tonnage curves comparing PbZn% between the 2012 and the 2013 undepleted Resource models for the Project in-situ deposit, all classifications, are shown in Figure 1.4 and Figure 1.5.
Figure 1.4 PbZn Grade-tonnage curves (Tonnes above cut-off versus Cut-off grade) for the in-situ Project deposit [Blue – 2012; Red – 2013]
1.6 Mining engineering study

The existing Namib underground mining inventory will be accessed through the refurbished South decline, from an existing portal on surface. The mining inventory in the North and N20 Lodes will be accessed from a separate North decline which will commence development in the pre-production period to gain early access to more mineable material.

Shrinkage and down-dip stoping by hand held air-leg and drill are the two mining methods proposed for the Project. Standard trackless diesel powered mobile equipment will be used in the underground operation for the footwall development and conventional stoping equipment will be utilised.

Geotechnical studies were undertaken by Celtis Geotechnical (Celtis), acting as sub-consultants to NRR, and peer reviewed by Snowden. The Celtis report describes expected operating conditions and ground support requirements at the Project. All geotechnical work was completed on the basis that the mine will be dry during operations as advised by NRR.
Ventilation requirements were estimated based on the proposed equipment fleet and mine design (e.g. drive dimensions). The ventilation design will use existing mined out stopes, were possible, to return air to surface. The planned Namib, North and N20 mine network is very simple with parallel declines and the shaft as intake between surface and the lowest mining level. The exhaust system is straightforward and comprises collection levels (possibly with connecting long hole raises) from the northern and southern sections, of the proposed mine, connected to the existing large void (old stope called the Junction stope) used as the main exhaust connected to surface with a long hole raise through the crown pillar. It is assumed that the main fans (2 by 75kW) will be installed on the surface above the Junction stope. As the Project presently has a limited mine life of less than 10 years, at a relatively low pressure, the main fans do not have to be designed for a long life and a high efficiency. The duty is suitable for axial flow fans to reduce overall costs by selecting a much simpler inlet box arrangement. The ventilation design is detailed in the ventilation study provided in Appendix B.

The underground mining and haulage is planned as an owner labour employed mining operation. The mining labour complement builds up to peak at about 191 people by 2016 from where it steadily declines again as indicated in Figure 1.6.

**Figure 1.6 Underground Labour Profile**

![Graph showing underground labour profile](image)

Source: Snowden, 2014

Underground production builds up from 2015 for 19 months when full production of 20 ktpm is reached in August 2016 as shown in Figure 1.7. A mining rate of 20 ktpm is sustained for approximately 22 months after which production declines due to limited mining inventory tonnage. Further exploration may allow the upgrading of current Inferred Mineral Resources and may also allow the addition of extensions at depth of the existing mineralised shoots.
1.6.1 Mining inventory

Snowden could not declare an Ore Reserve as the Life of Mine (LOM) design and schedule includes Inferred Mineral Resources. Table 1.6 presents the mining inventory Mineral Resource classifications for tonnages and grades in each resource category. The Indicated Mineral Resource constitutes about 65% of the total mining inventory, while the Inferred Resource is approximately 35%.

Table 1.6 Mining Inventory Categories

<table>
<thead>
<tr>
<th>Class</th>
<th>Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>425,151</td>
<td>3.38</td>
<td>3.00</td>
<td>5.71</td>
<td>51.2</td>
</tr>
<tr>
<td>Inferred</td>
<td>234,372</td>
<td>3.38</td>
<td>1.77</td>
<td>6.46</td>
<td>36.5</td>
</tr>
<tr>
<td>Total Inventory</td>
<td>659,523</td>
<td>3.38</td>
<td>2.56</td>
<td>5.97</td>
<td>45.95</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

1.6.2 Mining risk

Key risks identified in the study include the following:

- Lack of an Ore Reserve statement due the limited Mineral Resources
- Lack of available skilled labour, such as in-stope Rock Drill Operators (RDO), poses a risk to meet the targets in the required timeline.

1.7 Infrastructure and services

The engineering scope of work for the Project includes provision for surface and underground support services for the processing and mining operations.
All engineering design criteria are based on industry Codes of Practice, standards and procedures, South African National Standards (SANS) specifications and NRR specific documentation. In general, the required standards, procedures and legal compliance are adhered to, both for temporary use and permanent positioning and operation of buildings and equipment.

The process plant infrastructure and services include all administration, change room, ablution and general services buildings. Services include the provision of power, water, internal roads, sewerage and consumable storage.

Surface structures to support the mining include a compressor station, lamp room and crush, extension of the change house, offices for mining staff, mine maintenance building, medical facility, and parking area and covered bus stop shelter.

Services are provided for the mining and mining capital development which include compressed air, power, mine service water, potable water (inclusive of fire water) and return mine water.

1.8 **Environmental review**

NRR currently has in place an Exclusive Prospecting Licence (EPL), and submitted a mining licence application by 17 April 2014. An Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) was compiled by Colin Christian & Associates CC (CCA) as supporting documentation for the mining licence application, and submitted to the Ministry of Environment and Tourism (MET) on 17 December 2013.

The environmental review is found in Appendix D.

1.9 **Background and project history**

The mine operated for 24 years from 1968 to 1991. Underground levels were developed on a 30 metre (m) spacing with sublevels every 15 m. Historic production records indicate that 356,300 tonnes were milled at the old mine with yield grades of 5.3% zinc and 1.6% lead to produce 38,121 tonnes of zinc concentrate, and 14,142 tonnes of lead concentrate. Neither Snowden nor NRR have reviewed these records however, it is believed that approximately 1 Mt of ore has been extracted from the existing mine workings.

After being abandoned in 1992 the mining lease lapsed and after a period of time the site was vandalised and all surface infrastructure was damaged or removed. No remedial works or closure operations were undertaken prior to abandonment and the site was left in poor condition with rubbish, chemicals, mining equipment and unfenced open holes over the site.

In 2001 the EPL was granted to local geologists who primarily assessed the potential for re-processing the tailings dump and did not carry out any work on the remaining in-situ sulphide resources. No further work was completed until Kalahari Minerals Limited (‘Kalahari’) purchased 90% of the holding company and commenced work in 2007 which consisted of two drilling campaigns and testing extensions to the mineralised shoots.

Evaluation drilling of the surface tailings dumps was completed by Kalahari during September 2008 on a nominal 10 m by 10 m grid pattern totalling 178 reverse circulation (RC) holes for 1,450 m. Drilled depth to base of dump varied from 1 m to 15 m. Some access issues in the southwest and northwest corners of the dump prevented the completion of all planned holes.
RC drilling completed by Kalahari focused on delineating the North and N20 lodes only. Although some of the 2007 diamond holes were drilled beneath the South and the junction between the South and North Lodes by Kalahari. There was insufficient data to define a Resource in those zones.

In 2009 NRR took the project over from Kalahari.

NRR immediately began refurbishment of the underground mine. After dewatering, the mine was surveyed using a cavity monitoring system. Once the survey was completed, and survey control was established throughout the mine, a drilling programme commenced.

The mineralisation was mapped and a channel sampling program was completed which totalled 600 m of cut channel. An in house mineral resource estimate was completed by Kalahari (now owned by NRR) in 2009 this is shown in Table 1.7

<table>
<thead>
<tr>
<th>Area</th>
<th>Tonnes</th>
<th>Zn (%)</th>
<th>Pb (%)</th>
<th>Ag (ppm)</th>
<th>In (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailings Facility*</td>
<td>398,604</td>
<td>2.09</td>
<td>0.26</td>
<td>7.6</td>
<td>9.3</td>
</tr>
<tr>
<td>N20 &amp; North</td>
<td>611,903</td>
<td>6.45</td>
<td>2.27</td>
<td>42.1</td>
<td>31</td>
</tr>
</tbody>
</table>

Source: NRR, 2012
2 Underground mining study

2.1 Objective

The objective of this study is to develop a mine design and define equipment requirements, operating and capital cost estimates, ventilation and geotechnical requirements to assess the economic value of the Project in alignment with the NRR mining strategy. The objective is to deliver a Life of Mine (LOM) Plan to sustain a 250 ktpa processing plant.

The initial request for a MDP included the re-processing of tailings at a 300 ktpa until fresh material became available from underground mining. It was initially determined that the mining scheme would be developed on the larger tonnage Mineral Resources which are located at the North and N20 Lodes with the South being developed as mining progressed. As the tailings were excluded after the study commenced it was thought that immediately available fresh material from the Namib mine could accelerate the mining schedule and as such decided to include the re-development of the existing South decline in the pre-production period and limit the development of the North decline.

2.2 Mine design criteria

2.2.1 Purpose

The design criteria used in the design of the Project, as far as practicably possible, conforms to industry standards, current industry best practices and makes use of planning parameters by Snowden and NRR. In instances where no NRR planning parameters exist, benchmarked figures, established factors and industry parameters are used.

The mine design has been prepared with particular attention to zero harm to people and the environment. The Inferred and Indicated Mineral Resources were optimised, and comply with the strategic objectives of the Project and its stakeholders. However this mining inventory cannot be classified as a Mineral Reserve under the JORC 2012 code. The mining method is a combination of conventional shrinkage mining and conventional down-dip mining methods serviced by trackless footwall haulages and ramp systems.

2.2.2 Guidelines, codes and specifications

The mine design is conducted in accordance with the application of NRR specified guidelines which, not in all cases, are recommendations by Snowden. Notwithstanding the NRR specified guidelines, the Project is designed in accordance to applicable guidelines of the Namibian Ministry of Mines and Energy (NMME) and other statutory authorities. Where no such specifics or guidelines exist or were not readily available, South African standards, best practices and guidelines were used.

2.2.3 General design philosophy

The Project is designed according to the following objectives:

- To align the project within the strategic intent of NRR
- Equipment and components required for the project are standardised to NRR requirements where appropriate
- Use of understood technology
- Maximise opportunity for use of local labour
2.3 Mining strategy

Background

A scoping study was undertaken by NRR aimed at establishing the business case for the Project. The scoping study determined that a sustainable steady-state production rate of 250 ktpa could be achieved. The scoping study also concluded that at higher production rates the project becomes increasingly robust and would be able to weather greater levels of cost and metal price fluctuation. Furthermore, the overall level of capital expenditure is proportional to potential returns, in-line with an attractive mining investment, when considering the very low country risk.

The subsequent MDP undertaken by Snowden was to include the tailings retreatment until ore became available from underground mining. However it was determined from testwork undertaken at Mintek, the South African national mineral research organisation, that the recovery of zinc from the tailings was not economic and it was decided to exclude the tailings from the study. Snowden determined that an optimised throughput of 120 ktpa could be sustained from the Indicated Mineral Resources. This would allow for an extended LOM from subsequently discovered Mineral Resources from ongoing drilling. The NRR strategy required a throughput of 250 ktpa and it was then decided to include the Inferred Mineral Resources to create a mining inventory for this MDP.

2.3.1 Orebody characteristics

Underground mineralisation

Mineralisation at the Project has a general northwest trend. The mineralisation is defined by short strikes and extension at depth. Four major ore zones have been defined, namely the North, South, N20 and Junction (Figure 2.1), with well-developed gossans on surface. The mineralisation generally seems to conform to the marble, but can cut across lithologies. Where gossan is developed this generally comprises ferruginous material with some galena, cerussite and smithsonite. The gossans often extend to a depth of 10 m, while oxidation extends to 16 m. Fresh mineralisation comprises mainly sphalerite, galena, pyrrhotite and pyrite. The sphalerite is partly iron-rich due to the presence of marmatite (ZnFeS), an iron rich sphalerite containing more than 6% iron which is difficult to separate from pyrite and pyrrhotite. The marmatite level associated with the sphalerite is unknown. The occurrence of marmatite could have an impact on the iron content in the final zinc concentrate (LMS, 2012).
Dip

The typical dip of the shoots is between 45° and 90° with smaller volumes plunging at lower angles as indicated in Figure 2.2.

**Figure 2.2** Long section - dip orientation

Source: Snowden, 2014
A cut-off dip of 55° was selected as a base for the natural gravitation of blasted ore which is greater than the angle of repose for most rock types. The main reason for this is that the designed stopes of more 55° will ensure the natural gravitation of ore without any use of cleaning equipment. Table 2.1 shows the tonnage split per orebody zone with dip more than 55° and less than 55°, subdivided by Mineral Resource classification. The two dip regimes were used to define stopes amenable to gravity assisted cleaning shrinkage stoping and a mechanically assisted cleaning method such as down-dip mining.

### Table 2.1 Oredody dip per Mineral Resource category

<table>
<thead>
<tr>
<th>Dip (Degrees)</th>
<th>Category</th>
<th>Volume</th>
<th>Tonnes</th>
<th>Density</th>
<th>PB (%)</th>
<th>ZN (%)</th>
<th>AG (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;55</td>
<td>Indicated</td>
<td>74,652</td>
<td>257,549</td>
<td>3.45</td>
<td>3.16</td>
<td>7.11</td>
<td>50.09</td>
</tr>
<tr>
<td>&lt;55</td>
<td>Inferred</td>
<td>43,886</td>
<td>151,406</td>
<td>3.45</td>
<td>1.46</td>
<td>8.06</td>
<td>34.99</td>
</tr>
<tr>
<td>Sub Total</td>
<td></td>
<td>118,538</td>
<td>408,956</td>
<td>3.45</td>
<td>2.53</td>
<td>7.46</td>
<td>44.50</td>
</tr>
<tr>
<td>&gt;=55</td>
<td>Indicated</td>
<td>113,023</td>
<td>389,929</td>
<td>3.45</td>
<td>2.21</td>
<td>3.95</td>
<td>44.57</td>
</tr>
<tr>
<td>&gt;=55</td>
<td>Inferred</td>
<td>41,196</td>
<td>142,126</td>
<td>3.45</td>
<td>2.70</td>
<td>5.83</td>
<td>50.00</td>
</tr>
<tr>
<td>Sub Total</td>
<td></td>
<td>154,219</td>
<td>532,055</td>
<td>3.45</td>
<td>2.35</td>
<td>4.45</td>
<td>46.02</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

The Mineral Resource tonnage indicates that 43% of the orebody has a dip angle of less than 55°, while 57% of the orebody dips at an angle of more than 55°. This analysis formed the basis for the selection of the mining method.

### Strike and width

The Lead-Zinc mineralised mining true width ranges from 2.52 m to 13.59 m with an average width of approximately 5.88 m. Similarly, the strike extent of the orebody ranges from 9.62 m to 91.24 m, averaging a strike extent of approximately 24.91 m.

### Depth

The topography surrounding the mining area is fairly flat. The resource cut-off depth for consideration in this study is 58 meters below surface (mbs) to 318 mbs. Figure 2.3 shows the depth extent of the orebody below surface.
Structure

TECT Consulting (TECT) undertook a structural geological study of the project area. The main observations from the study are summarised as follows:

- There is a distinct relationship between the trends of fold limbs, northeast to southwest trending cleavage and shear zones.
- A cone axis plunging at 48° → 187° with a cone angle of 70° including the spread of poles to compositional banding. The orientation of cataclasite and other shears is similar; suggesting that intra limb shearing took place.
- Biotite/quartzite banding is oriented more or less vertically and strikes north-south.
- Sub-vertical magnetite–hematite veins strike towards 030° which is different from the orientation of most ore shoots.
- Ore shoot orientations overlap the distribution of fold axis orientations. Old axes tend to be about 15° shallower than ore shoots, suggesting that mineralisation may not occur ubiquitously in fold hinges. Basson and Tennant (2012) interprets the mineralisation to be controlled by two structural orientations. The first is parallel to the regional fold hinge zone and the second steeper but sub-parallel to the fold axes. This is likely to be an intersection lineation between fanning cleavage and lithological banding (Basson and Tennant, 2012).
- The upper mined out part of the ore body is associated with brittle-ductile deformation, whereas that deeper more sulphide-rich disseminated mineralisation is relatively in-situ fold controlled material.
- Hornblende-tourmaline-biotite pegmatite bodies which pre-existed mineralisation would have had a damming effect on mineralising fluids and exerted some control on the development of fractures and other fluid pathways.

Source: Snowden, 2014
Grade distribution

Figure 2.4 to Figure 2.6 illustrates grade distributions of Zn (%), Pb (%) and Ag (g/t) respectively. High Zn grade is indicated on the North zones, while Pb and Ag grades are fairly evenly distributed across the North and South areas.

Figure 2.4  View looking NE – Zinc (%) grades

Source: Snowden, 2014

Figure 2.5  View looking NE – Lead (%) grades

Source: Snowden, 2014
Rock strength

Strength studies were undertaken by Celtis Geotechnical cc (Celtis) using samples tested by Rocklab in Pretoria, South Africa.

The most common rock type in the rock mass is marble with interbedded calc silicates and calc silicates with interbedded marble. These and the ore body were tested for Uniaxial Compressive Strength (UCS), Density, Elastic Modulus and Poisson’s Ratio.

The results are shown in Table 2.2.

Table 2.2  Rock type desnsity and strength

<table>
<thead>
<tr>
<th>Borehole (ID)</th>
<th>Depth (m)</th>
<th>Rock Type</th>
<th>Density (g/cm$^3$)</th>
<th>Strength (UCS) (Mpa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>NLDDK001</td>
<td>12.64 - 12.81</td>
<td>Orebody</td>
<td>3.52</td>
<td>143.5</td>
</tr>
<tr>
<td></td>
<td>12.81 - 13.04</td>
<td></td>
<td>3.88</td>
<td>134.4</td>
</tr>
<tr>
<td></td>
<td>161.32 - 161.65</td>
<td></td>
<td>2.73</td>
<td>64.6</td>
</tr>
<tr>
<td>NLD013</td>
<td>161.65 - 161.92</td>
<td>Marble interbedded with Metacalc Silicate</td>
<td>2.71</td>
<td>62.7</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2.71</td>
<td>56.8</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2.80</td>
<td>146.3</td>
</tr>
<tr>
<td>NLD013</td>
<td>137.56 - 137.87</td>
<td>Metacalc Silicate</td>
<td>2.78</td>
<td>142.6</td>
</tr>
<tr>
<td></td>
<td>137.87 - 138.20</td>
<td>Interbedded with Marble</td>
<td>2.86</td>
<td>144.1</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2.86</td>
<td>102.2</td>
</tr>
</tbody>
</table>

Source: Celtis 2014
The difference in UCS between the marble and the meta calc silicates is not observed in the core or underground. The rock types are so intimately interbedded that the rock mass constitutes both rock types. Furthermore, the rock mass quality of the ore body and country rock is sufficiently good that the mining of each ore shoot will be stable without permanent support.

### 2.3.2 Mining method selection

The key drivers in determining a method are:

- Ore body and geotechnical characteristics i.e. mainly orebody dip
- Minimum capital investment
- Low production rates
- Flexible to ensure minimum waste development
- Minimise dilution / maximise net profit per tonne
- Proven method and technologies.

The selection of shrinkage stoping and down-dip mining methods was mainly based on orebody characteristics. For the purposes of this study, shrinkage stoping is attributed to stopes dipping at greater than 55°, while the down-dip method is based on the orebody section with dip less than 55°. However these mining methods have their own disadvantages, namely:

- Labour intensive
- Requirement for experienced and skilled jackleg miners
- Working conditions difficult and relatively hazardous
- Slow ramp-up to steady state production
- High footwall development rates are required to maintain steady state production.

NRR decided not to undertake a trade-off study of mining methods.

**Shrinkage stoping**

Due to the steep dip and the competent nature of the orebody and surrounding rockmass, a conventional shrinkage stoping method was selected as a preferred extraction method supported by mechanised footwall infrastructure.

The strike length of the shoots averages 24.25 m, and where the strike extent is greater than 40 m, adjacent stopes will be developed leaving 8 m sill pillars on strike between the stopes.

Shrinkage stoping is an overhand mining method that relies on broken material being left in the stope to be used as the “working floor” and to support the walls (Figure 2.8). The volume of the broken ore will swell by 30% to 40% due to blasting. Approximately 40% of the blasted ore will be withdrawn and when the stope reaches the upper limits, the remaining 60% of broken ore is recovered.
Figure 2.7 illustrates a typical shrinkage stoping with ore drive configuration. The shrinkage stoping will be accessed via 1.8 m (W) by 2.1 m (H) travelling ways installed with a ladder-ways and supported with timber laggings. Each shrinkage stope will consist of a timber bay, travelling ways, ore and loading drives to facilitate loadings and hauling of ore and the free movement of material and personnel into the stoping areas. Ventilating air will be directed to the stopes via access travelling ways developed either side of the shrinkage stope to the level above where the air will be directed to the return airway. Section 3.3.2 in the geotechnical chapter of this report provides further comment on in-stope safety requirements.

Ore is excavated in horizontal slices from the bottom-up and stopes will be mined at variable slices of 0.8 m to 1.5 m. Blast holes will be drilled using three (3) Rock Drill Operators (RDO) operating hand held pneumatic drilling machines and drilling both sides of the stopes with an effective hole length of 1.2 m. The planned effective advance per blast is 1.0 m/blast. Blast holes will be charged to three quarters of their length with commercial explosives and initiated with non-electric shock tube detonators. The production per 1.0 m advance from a blasted panel is estimated at 259 tonne (t), assuming a drilling productivity per RDO of 25 m³/RDO/shift.

The working face will be supported during drilling and charging-up activities with temporary mechanical active support jacks (typically cam-lock jacks or acrow props). These temporary jacks will be used in conjunction with nylon netting supported between the temporary jacks to prevent any blocks of ground that may come loose during the drilling and charging-up of the stope face. Where ground conditions deteriorate, an issue based risk assessment should be performed by the appointed competent geo-technical engineer who shall specify additional support or remedial action that should be taken to ensure the safety of persons working under overhanging working faces.

*Source: Snowden, 2014*
**Down-dip mining**

Stoping activities on each level will consist typically of a raise development, ledging and equipping a raise. Panel lengths will vary depending on the strike distance, but it envisaged that the panel length will not be more than 20 m on average.

A typical down-dip stope will consist of the travel way, ore drive, raise, timber bay and loading bay as indicated in Figure 2.9.
The panels will be drilled using hand held pneumatic drills and blasted on dayshift, and the broken ore will be removed on the night shift. It is assumed that each crew achieves 20 blasts per month at 1.0 m advance per blast. Winches will be used to clean the panels, gullies and raises to ore passes or directly into the ore drives below depending on the stoping configuration.

Wide ledging commences immediately after the raise is holed. The stoping phase will form a 70° configuration from the raise to allow the ease of breaking a face and to allow effective cleaning into the raise as indicated in Figure 2.10.
Equipping crews will commence equipping of stope panels with electrical boxes, cabling, water pipes, air pipes, winch beds, winches, scrapers, ropes, snatch blocks, a mono-winches and blasting barricades.

Ore is mined in slices from the top-down at a variable height from 0.8 m to 1.5 m. After making safe-procedures and the installation of temporary support, roofbolts will be drilled and installed on a 2 m by 2 m pattern in the hanging-wall. The temporary support shall consist of mechanical active support jacks (typically cam-lock jacks or acrow props). These temporary jacks will be used in conjunction with suitable barricades supported by the temporary jacks to prevent any falling and / or rolling rocks that may come loose during the drilling and charging-up of the stope face. Where ground conditions deteriorate, an issue based risk assessment should be undertaken by the appointed competent geotechnical engineer who shall specify additional support or remedial action that should be taken to ensure the safety of persons working on steeply inclined working faces.

Following the installation of the temporary support and permanent roofbolts, blast holes, with an effective hole length of 1.2 m, will be drilled by three (3) RDO’s, operating hand held pneumatic drilling machines either side of the ore raise. Blast holes will be charged to three quarters of their length with commercial explosives and initiated with non-electric shock tube detonators at the end of the shift. The production per 1.0 m advanced from a blasted panel is estimated at 259 t per shift based on a drilling productivity per RDO of 25 m³/RDO/shift.
Following blasting fume clearance, the night shift crews enter the down-dip stope and follow the “making-safe” procedure. Scraper winches (37kW) will be rigged up and begin cleaning the blasted ore from the stope face into the raise where the ore will be moved by the raise scraper winch (37kW) into an ore drive located at the bottom of the raise. The broken ore at the bottom of the raise will be mucked using the LHD into awaiting trucks for transport to the surface processing plant.

2.3.3 Mining unit layout

Primary access development

Current underground infrastructure

The mine operated for approximately 24 years with ore being mined to a depth of 210 m below surface. There are currently two access declines to the South and the North of the ore zones. The South decline is about 1.45 km long and accesses the South and Junction ore zones, and the North decline is about 150 m long and accesses the North ore zone. The existing development and stoping is shown in Figure 2.11.

Figure 2.11 Existing development and stoping

Source: Snowden 2013

Junction and South decline

During the site visit conducted by Snowden from 18 September 2013 to 20 September 2013 and an additional site visit from 7 May 2014 to 9 May 2014, a visual inspection showed the South decline to be 2.1 m high by 2.4 m wide at its narrowest point, with an average cross-sectional area of 6.06 m².
The decline gradient is estimated to be as steep as 1 in 4 in some areas and Snowden considers that the gradient is too steep for four wheel dump trucks. The decline has been constructed around and through the ore shoots often very close to stopes and in one case breaks into a stope. These stoping voids occur beside both walls and under the floor in some cases. Special precautions and measures are required for the safe sliping of the South decline to the required final dimensions of 3.2 m (W) by 2.5 m (H).

Corners in most of the South decline sections are very tight and will require scaling off to allow the selected trucks to manoeuvre without vehicle damage (two such corners will require significant modifications as indicated in Figure 2.1. The footwall road is undulating due to poor past mining practice and will require significant grading and running surface construction.

**Figure 2.12 Illustration of South decline Sliping**

Some of the drives and ramps that could have been developed straight have kinks and will require scaling off for alignment.

The South decline dimensions need to be increased by either scaling or blasting process in order to fit the selected load and haul equipment.

The existing shaft will be used to accommodate the underground services which includes:

- compressed air
- mine water
- electrical power
- communications
- other as required.
Namibian mining legislation does NOT prescribe a minimum acceptable clearance required for mechanised mobile mining machinery in haulage-ways and maintains that the mine should ensure that clearances are maintained that are safe for persons and equipment. NRR has prescribed a minimum clearance of 0.5 m either side of the largest dimension of the largest machine. Planned maximum machine dimension for the South decline is the EIMCO 913 loader with an operating width across the bucket of 2.15 m. The minimum dimensions selected for the South decline were therefore 3.2 m (W) by 2.5 m (H).

Snowden accepts that there may be no legal requirement for minimum vehicle clearance for underground vehicles but recommends vehicle clearances in the range from 0.75 m to 0.9 m either side of the maximum operating width for safe operation. As such, the vehicle clearances used in the design have been recorded in the risk register as a higher risk than would normally be indicated for vehicles travelling in haulage-ways.

### Table 2.3 Sliping Requirements for the South decline (3.2 mW by 2.5 mH)

<table>
<thead>
<tr>
<th></th>
<th>Length ( (m) )</th>
<th>Excavated Volume ( (m^3) )</th>
<th>Final Volume ( (m^3) )</th>
<th>Blasted Volume ( (m^3) )</th>
</tr>
</thead>
<tbody>
<tr>
<td>South Ramp (Existing)</td>
<td>1,461</td>
<td>8,855</td>
<td>11,682</td>
<td>2,827</td>
</tr>
<tr>
<td>South Ramp (New)</td>
<td>98</td>
<td>784</td>
<td>784</td>
<td>784</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014 (based on NRR requirements)

The decline gradient will also need to be flattened to 1:6 (maximum 1:5) by either scaling or backfilling with waste rock material to prepare an acceptable gradient for the truck hauling of broken rock via the ramp system to surface operations.

The top 70 m of the South decline is steep (1 in 4) and located in poorer ground conditions as indicated in Figure 2.13. Snowden is of the opinion that the decline portal may need to be moved to a suitable position or supported according to geotechnical recommendations following an on-site geo-technical risk assessment of the rock conditions.
Figure 2.13  Existing South decline boxcut and portal

The shaft access and ladderways on Level 3 are shown in Figure 2.14. These access-ways would previously have been used to move men and air around the mine.

Figure 2.14  Level shaft access and ladderways

Source: Snowden, 2014

North decline

The North decline has similar dimensions to the South decline. It is located in good structural conditions but may need to be supported following an on-site geotechnical risk assessment. Figure 2.15 shows the North decline portal entrance to the Northern working areas. The North decline consists of a single level and a small, historically mined up-dip stoping void.
NRR has prescribed a minimum clearance of 0.7 m either side of the largest dimension of the largest machine. Planned maximum machine dimension for the North decline is the Fermel LHD 7 t loader with an operating width across the bucket of 2.5 m. The minimum dimensions selected for the North decline are therefore 3.9 m (W) by 2.6 m (H).

Snowden accepts that there may be no legal requirement for minimum vehicle clearance for underground vehicles but recommends vehicle clearances in the range from 0.75 m to 0.9 m either side of the maximum operating width for safe operation.

Table 2.4  Sliping Requirements for the North decline (3.9 mW by 2.6 mH)

<table>
<thead>
<tr>
<th></th>
<th>Length (m)</th>
<th>Excavated Volume (m³)</th>
<th>Final Volume (m³)</th>
<th>Blasted Volume (m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>North Ramp (Existing)</td>
<td>186</td>
<td>1,308</td>
<td>1,884</td>
<td>576</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014 (based on NRR requirements)

Figure 2.15  North decline portal

Source: Snowden, 2014

2.3.4 Access methodology

Decline template

The ramp system will reach nine levels approximately 220 m below surface. No refrigeration is required for the extent and depth of the ultimate expected mine excavation. The average vertical distance between levels is approximately 25 m. It is proposed to utilise the existing North decline and re-position the South portal to an appropriate position that will allow for acceptable ramp gradients. The dimensions of the North and South decline will be 3.9 m (W) by 2.6 m (H) and 3.2 m (W) by 2.5 m (H) respectively to allow access for trackless equipment, underground personnel, material and rock transportation.
Gradient

The gradient of the decline is variable with the constraints being capital cost of developing the vertical distance and the ability for productive haulage from the decline. Operationally, 1.6 to 1.7 is internationally accepted as the steepest practical gradient while still including curves and allowing for safe stoppage of machines on the down slope. It is proposed that the gradient for the declines should be 1:6 along the centreline throughout and flattened around the turning points.

Level drives should be 1:50 up (where possible) to allow for water drainage and ease of drilling/mucking operations. Where a choice between incline versus decline existed the incline was chosen for the consideration of handling of water (self-draining) and the control of muck near the brow.

Radius of curvature

The minimum turning radius was considered for all possible trucks that can be used in the decline and is in the range 7 m to 10 m. The radius of plus 10 m provides operational capability without causing attrition of the driveline through working at full lock.

South decline positioning

Due to the extreme steep gradient of the upper section of the South decline, the top 200 m will need to be replaced. The new portal is located adjacent to the hill containing the shaft and across from the proposed process facility as shown in Figure 2.16.

Figure 2.16  Proposed new portal position

Source: Snowden, 2014
Figure 2.17  South portal repositioning

North decline positioning

The existing North decline (Figure 3.6) will be used as the gradients are acceptable for trackless truck haulage. Re-use of the North decline will require rehabilitation involving the scaling off to increase the size of the first 186 m to 3.9m (W) by 2.6m (H) and rehabilitation of the running road surface.
2.3.5 Primary footwall development template

The primary footwall development consists of trackless development of the declines and main drive levels. Level access drives are developed from the decline with a downward gradient of 1 in 50 to an established sump then upward gradient of 1 in 50 to ensure water is kept away from the decline in order to reduce roadway degradation. The lengths of the level access drives to the ore drives are typically designed to average 10 m, with multiple crosscut drives accessing the individual ore-shoots. A typical footwall development will consist of access drive, main haulage, ore drive, loading bays and refuge bays as shown in Figure 2.19.
The main haulages will be 3.9 m (W) by 2.6 m (H) in size and from these haulages, travelling ways will be developed at 34° to access the ore-shoot.

2.3.6 Primary reef development template

Shrinkage stoping

Primary reef development will consist of the travelling ways and two end travelling ways, developed at 1.8 m (W) by 2.1 m (H), up to the level above to provide access and the ventilation to the stope.

Down-dip mining

Similarly to shrinkage stoping, primary reef development consists of travelling ways, raises, and winch chambers. The travelling ways developed to the ore-shoots are 1.8 m (W) by 2.1 m (H) and the dip raises are developed conventionally at 1.5 m (W) by 2.1 m (H). On-reef development will be drilled with hand held pneumatic rockdrills and cleaned with scraper winches. Standard raises will be 25 m long on the vertical height between levels.

2.3.7 Mining operation

Operating hours

Annual calendar hours, operating hours and productive hours were calculated based on the work roster, shift lengths, equipment availability and expected delays.

The operating cycle for the underground mining operations is a five (5) day week.

Based on local requirements, the operation will work five (5) days per week. The overlap between maintenance and downtime has been allowed for in the cost model. Table 2.5 shows a four week proposed work roster cycle.
Table 2.5  Typical Monthly roster cycle

<table>
<thead>
<tr>
<th>Week</th>
<th>Day</th>
<th>Shift 1</th>
<th>Shift 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Monday</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Week 1</td>
<td>Tuesday</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Wednesday</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Thursday</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td></td>
<td>Friday</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td></td>
<td>Monday</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Week 2</td>
<td>Tuesday</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Wednesday</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Thursday</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td></td>
<td>Friday</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td></td>
<td>Monday</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Week 3</td>
<td>Tuesday</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Wednesday</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Thursday</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td></td>
<td>Friday</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td></td>
<td>Monday</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Week 4</td>
<td>Tuesday</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Wednesday</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Thursday</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td></td>
<td>Friday</td>
<td>5</td>
<td>5</td>
</tr>
</tbody>
</table>

Note 1: A third shift allows for the LHD and Truck operators to fully utilise the equipment. Cleaning and hauling of broken ore between the blasting times may take place on the provision that no persons enter into the blasted areas or operate the equipment in the return airways due to the risk of “gassing” from blasting fumes.

Table 2.6 below shows the calendar used for planning and scheduling based on number on days a year excluding public holidays and Christmas break.

Table 2.6  Production Cycle breakdown per calendar year

<table>
<thead>
<tr>
<th>Activity</th>
<th>Days</th>
</tr>
</thead>
<tbody>
<tr>
<td>Days per annum</td>
<td>365</td>
</tr>
<tr>
<td>Public holidays</td>
<td>9</td>
</tr>
<tr>
<td>Saturdays</td>
<td>52</td>
</tr>
<tr>
<td>Sundays</td>
<td>52</td>
</tr>
<tr>
<td>Working days</td>
<td>261</td>
</tr>
<tr>
<td>Months</td>
<td>12</td>
</tr>
<tr>
<td>Working days per month (average)</td>
<td>22</td>
</tr>
<tr>
<td>Blast strike rate (10% misfires, breakdowns etc.)</td>
<td>2</td>
</tr>
<tr>
<td>Average Combined shifts per month (2 Shift per day)</td>
<td>44</td>
</tr>
</tbody>
</table>

Shift duration is 9 hours clock-in to clock-out. Table 2.7 below details the non-face time hours accumulated during a normal shift.
## Table 2.7 Hours Lost per Shift-Updated

<table>
<thead>
<tr>
<th>Fixed Shift Delays</th>
<th>Down-dip</th>
<th>Shrinkage</th>
<th>Development</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>Change house down</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Hours</td>
</tr>
<tr>
<td>Travel down</td>
<td>0.5</td>
<td>0.5</td>
<td>0.5</td>
<td>Hours</td>
</tr>
<tr>
<td>Safety and inspection</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
<td>Hours</td>
</tr>
<tr>
<td>Meal break</td>
<td>1.0</td>
<td>1.0</td>
<td>1.0</td>
<td>Hours</td>
</tr>
<tr>
<td>Make area safe</td>
<td>0.5</td>
<td>0.5</td>
<td>0.5</td>
<td>Hours</td>
</tr>
<tr>
<td>Clean, stow and re-fuel</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
<td>Hours</td>
</tr>
<tr>
<td>Travel up</td>
<td>0.5</td>
<td>0.5</td>
<td>0.5</td>
<td>Hours</td>
</tr>
<tr>
<td>Blasting</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Hours</td>
</tr>
<tr>
<td>Other</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>Hours</td>
</tr>
<tr>
<td><strong>Total lost hours per shift</strong></td>
<td><strong>3.0</strong></td>
<td><strong>3.0</strong></td>
<td><strong>3.0</strong></td>
<td><strong>Hours</strong></td>
</tr>
</tbody>
</table>

### Development cycle

#### Preparation

At the beginning of each shift (regardless of mining activity) the face preparation procedure will consist of the following activities:

- A safety briefing
- Early entry examination procedures
- Water-down the face area
- Make safe using temporary support (where necessary)
- Bar-down loose rock.

After the area has been declared “safe” the miner may authorise the mining activity (preparation, cleaning, support and drilling) to take place. The miner demarcates any geological discontinuities according to the mine standards.

#### Support

Once cleaning of the development drives is complete the responsible person or miner examines the area again and marks-off the support. Spot bolting when required will be the only support using 1.8 m split sets. According to the geotechnical report (James JJ, 2014) no systematic support is required.

The following support cycle will be put in place:

- Hangingwall bolt spacing : 3.0 m
- Support ring spacing : 3.0 m
- Hangingwall bolts per ring : 1 bolt
- Bolts per hours : 7 bolts per hour
- Time to support round : 15 minutes (including installation and setup).
Drilling

The responsible person will enter the working place at the beginning of each shift and carry out a proper examination to ensure that the area is safe. The area will be temporarily made safe by barring down of loose rock and washing-down of the immediate area. Following support drilling and installation, the face will be examined for misfires. Misfires will be dealt with in accordance with the Standard Operating Procedure (SOP), and every socket on the face washed out and plugged with the socket plugs. When all the sockets are washed the face will be marked off for drilling according to the appropriate blasting pattern. Drilling will then commence.

The following holes will be drilled (3.9 m (W) by 2.6 m (H)):

- Pilot holes: 2 holes by 1.5 times standard shot hole length
- Burn holes: 5 holes by 2.8 m long
- Reamed holes: 0 holes by 2.8 m long
- Easer holes: 53 holes by 2.8 m long
- Smooth wall holes: 0 holes by 2.8 m long
- Total holes: 60 holes by 2.8 m long

Total hole length to be drilled during a blasting shift (per standard ore drive):

- Pilot holes: 8.4 m
- Burn holes: 14 m
- Reamed holes: 0 m
- Easer holes: 147 m
- Smooth wall holes: 0 m
- Total Holes: 169.4 m

The average cycle time for to complete the drilling of the footwall drives is 6.0 face hours.

Blasting

The shot holes will be charged with a shock tube detonator 8D inserted into a primer cartridge at the bottom of the hole. Pre-packaged capsule type explosives will be inserted into the hole and tamped using a charging stick. Tamping will be required with the use of cartridge explosives. Blasting will take place at the end of the shift from a central point determined by the mine manager.

The average cycle time for to complete charging-up, connecting and timing of the shot holes in the development end is expected to be 1.0 face hours.

Cleaning

The blasted end will be cleaned on the shift following the blast. When the re-entry period has expired the miner or responsible person will enter and wash down the area and make safe prior to cleaning.

The average cycle time (load and haul) to clean a flat development end face is approximately 2.5 hours.
**Stoping cycle**

Labour and production planning is based on two 9 hour production shifts during each of 22 shifts per month (5 day a week calendar). In order to facilitate fixed time blasting, each production shift will be followed by a one half hour re-entry period minimum. It is envisaged that the stoping cycle will start with the morning shift (06h00 to 15h00). After the initial waiting place procedure the crew will enter the stope, make safe according to standards and procedures and start to prepare the cleaned panel for the drilling cycle.

These cycles entail the installation of temporary support (where applicable) and face preparation for marking of the drill holes.

The Rock Drill Operators (RDO’s) prepare their hand held pneumatic drilling equipment whilst the face is being marked. The drilling cycle normally peaks at 09h00 and finishes at 13h30, allowing sufficient time to charge the panel. Rigging of the face scraper and raise scraper will be done on dayshift in order to ensure that the nightshift will only have to concentrate on cleaning.

Blasting will take place from 16h00, followed by the re-entry period. During this period, material can be transported into the workings on the intake areas of the mine and routine maintenance of equipment can be carried out.

Night shift is planned to start at 18h00 and end at 03h00. Panels and development that have been drilled and charged will be blasted at 04h00 following which a re-entry period will be observed.

**Steady state ore production**

At steady state, ore production targets will be achieved from two (2) shrinkage stopes and two (2) down-dip mining stopes. Two stopes for each of the methods will be blasted on a shift, on the second shift these will be cleaned from the blasting activities on the previous shift. Drilling, blasting and cleaning activities for ore production and development will take place on the morning and night shifts in order to obtain the 20 ktpm production rate.

The following activities are required for the stoping operations:

- on reef development e.g. travelling ways
- raise development
- equipping
- stoping.

The steady state Basic Mining Equations (BME’s) for both shrinkage and down-dip stoping are presented in Table 2.8. The input mining parameters are identical for each decline, resulting in marginally different tonnage outputs as a result of the variance in resource/planned stoping widths between the two declines. It must be noted that in most instances the shrinkage and down dip stopes will be mined concurrently which will produce approximately 20 ktpm.
### Table 2.8 Basic Mining Equation for steady state production

<table>
<thead>
<tr>
<th>Basic Mining Equation - NRR</th>
<th>Shrinkage</th>
<th>Down-dip</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>Panels per raise line</td>
<td>1</td>
<td>2</td>
<td>no.</td>
</tr>
<tr>
<td>Blasting Shifts</td>
<td>2</td>
<td>2</td>
<td>no.</td>
</tr>
<tr>
<td>Crews per developing raise line</td>
<td>1</td>
<td>1</td>
<td>no.</td>
</tr>
<tr>
<td>No. of stope raise lines</td>
<td>2</td>
<td>2</td>
<td>no.</td>
</tr>
<tr>
<td>No. of development raise lines</td>
<td>2</td>
<td>2</td>
<td>no.</td>
</tr>
<tr>
<td>Average face length</td>
<td>20.0</td>
<td>20.0</td>
<td>m</td>
</tr>
<tr>
<td>Average mining width (maximum single cut)</td>
<td>2.0</td>
<td>2.0</td>
<td>m</td>
</tr>
<tr>
<td>Dilution</td>
<td>5%</td>
<td>5%</td>
<td>%</td>
</tr>
<tr>
<td>Stope ore withdrawal percentage</td>
<td>40%</td>
<td>100%</td>
<td>%</td>
</tr>
<tr>
<td>Effective stoping width</td>
<td>2.10</td>
<td>2.10</td>
<td>m</td>
</tr>
<tr>
<td>Effective drill steel length</td>
<td>1.2</td>
<td>1.2</td>
<td>m</td>
</tr>
<tr>
<td>Advance per blast</td>
<td>1.00</td>
<td>1.00</td>
<td>m</td>
</tr>
<tr>
<td>Advance efficiency</td>
<td>83%</td>
<td>83%</td>
<td>%</td>
</tr>
<tr>
<td>Resource S.G.</td>
<td>3.45</td>
<td>3.45</td>
<td>t/m³</td>
</tr>
<tr>
<td>Immediate footwall S.G.</td>
<td>2.6</td>
<td>2.6</td>
<td>t/m³</td>
</tr>
<tr>
<td>Bulking factor</td>
<td>65%</td>
<td>65%</td>
<td>%</td>
</tr>
<tr>
<td>Reef broken S.G.</td>
<td>2.24</td>
<td>2.24</td>
<td>t/m³</td>
</tr>
<tr>
<td>Blasts per month stoping</td>
<td>20</td>
<td>20</td>
<td>blast/mo</td>
</tr>
<tr>
<td>Stopping blast efficiency</td>
<td>75.5%</td>
<td>75.5%</td>
<td>%</td>
</tr>
<tr>
<td>m³ per RDO per shift</td>
<td>12</td>
<td>12</td>
<td>no.</td>
</tr>
<tr>
<td>No. of RDOs per crew</td>
<td>3</td>
<td>3</td>
<td>no.</td>
</tr>
<tr>
<td>m³ per stope panel blast per shift</td>
<td>36</td>
<td>36</td>
<td>m³</td>
</tr>
<tr>
<td>m³ per stoping crew per month</td>
<td>720 (x2)</td>
<td>720 (x2)</td>
<td>m³</td>
</tr>
<tr>
<td>Stope m³ per month</td>
<td>2,880</td>
<td>2,880</td>
<td>m³</td>
</tr>
<tr>
<td>Tonnes per month</td>
<td>9,936</td>
<td>9,936</td>
<td>t</td>
</tr>
<tr>
<td>Tonnes per stope raise line per day (average)</td>
<td>452</td>
<td>452</td>
<td>t</td>
</tr>
<tr>
<td>Tonnes per stope raise line per day (maximum)</td>
<td>497</td>
<td>497</td>
<td>t</td>
</tr>
<tr>
<td>Reef development tonnes per month</td>
<td>912</td>
<td>761</td>
<td>t</td>
</tr>
<tr>
<td>Decline tonnes per month</td>
<td>10,848</td>
<td>10,697</td>
<td>t</td>
</tr>
<tr>
<td>Tonnes per level per day</td>
<td>493</td>
<td>486</td>
<td>t</td>
</tr>
</tbody>
</table>

### 2.4 Mining equipment and logistics

#### 2.4.1 Mining equipment

The North decline development and the Return Air Way (RAW) will be developed with trackless machinery with hand-held pneumatic blast-hole drilling. The fleet at each decline, at steady state, will consist of an LHD and a matching number of dump truck servicing the LHD. Additional trackless machinery will comprise two Light Delivery Vehicles (LDVs), one tele-handler / integrated tool carrier and one material transport (Utility Vehicles) UVs.
The waste rock will be removed with an LHD loaded into a 14 tonne truck. The waste rock will be trammed to the previously mined out areas and dumped into the existing voids as backfill where possible. Loading into the diesel trucks with the LHD will be via the loading bays located on each level.

The stoping equipment will consist of S215 pneumatic rock drills together with scraper winches (in the case of down-dip method) and ancillary pneumatic tools will comprise the main equipment in the stopes. The ore will be removed from the ore drives using a 6 t LHD loading into a 14 t truck hauled up the main decline into the surface stockpiles.

Table 2.9 lists the equipment fleet used in this study. A brief description of each item is presented in the following section. The mining equipment listed in Table 2.9 does not include swing units.

### Table 2.9 Underground equipment fleet summary

<table>
<thead>
<tr>
<th>Application</th>
<th>Equipment</th>
<th>Manufacturer</th>
<th>Description</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Loading and Hauling</td>
<td>Liberator 14t</td>
<td>Fermelsa</td>
<td>Truck</td>
<td>4</td>
</tr>
<tr>
<td></td>
<td>T-mech750</td>
<td>Fermelsa</td>
<td>LHD</td>
<td>2</td>
</tr>
<tr>
<td>Development Drilling</td>
<td>S25 Rockdrill</td>
<td>SECO</td>
<td>Hand held drill rig</td>
<td>12</td>
</tr>
<tr>
<td>Stoping</td>
<td>S215 Rockdrill</td>
<td>SECO</td>
<td>Hand held drill rig</td>
<td>24</td>
</tr>
<tr>
<td>Raising</td>
<td>S215 Rockdrill</td>
<td>SECO</td>
<td>Hand held drill rig</td>
<td>8</td>
</tr>
<tr>
<td>Ancillary</td>
<td>UV80</td>
<td>Aard</td>
<td>UV</td>
<td>1</td>
</tr>
</tbody>
</table>

### Loaders

A Fermel T-Mech 750 was selected for the study, as it offers high stope productivity and is still suitable for cleaning development headings. The technical specifications of the loader are provided in Figure 2.20.
For the purposes of this study, a Fermel Liberator 15 t truck was used. This truck offers greater manoeuvrability than other types and can be loaded by a T-Mech loader in approximately two passes. The specifications and dimensions of a Liberator 15 t truck are displayed in Figure 2.21.

**Trucks**

Figure 2.21  Liberator 15 t truck
Development drilling

Conventional hand-held pneumatic jacklegs will be used for face drilling development rounds. For the purposes of this study, a SECO S25 development jackleg has been selected and is capable of drilling the required cut length of 2.5 m.

Stope drilling

S215 Pneumatic rock drills, together with scraper winches and ancillary pneumatic tools, will comprise the main equipment in the stopes. The ore broken for down-dip stopes will be scraped to the raiseline ore passes using scraper winches (37kW units in raises, 37kW units for stope faces) where applicable. In the case of shrinkage stoping, where the ore shoots dip allows, the ore will gravitate towards the ore drives where loading will take place.

Figure 2.22 S215 hand held pneumatic stope drill

Ancillary equipment

A UV80 (Utility Vehicle) will be used as ancillary equipment for this study.

2.4.2 Materials handling

Material will be delivered to the mine on low-bed trucks and will be stored for loading onto the UV and driven down the decline to the required level. Small materials, explosives and explosive accessories will be transferred into individual operating sections on the level.

Equipment such as pipes and winches will be stored on surface and will be transferred to underground sections when needed.
2.4.3 Water handling

Water discharged from the mining operations or from seepage will be controlled at source, where possible, and pumped to a drainage system where the dirty water will be allowed to gravitate to a collection sump located at the bottom of the ramp system. Water from the stoping horizon will be allowed to gravitate to the crosscut below and collected in a sump equipped with a dirty water pump. From here it will be pumped with footwall development water to a sump and from these sumps, the Mine Return Water (MRW) will drain to a main sump located at the bottom of the ramp system. The larger particles will be allowed to settle in the sump before the water is pumped to a main pump chamber for pumping to surface. The main sump located at the bottom of the ramp system will be cleaned of grit and sludge using the LHD and diesel trucks. This service water will then be pumped to the processing plant for recycling.

Drinking water will be provided at the ramp access located on each level.

2.4.4 Trackless vehicle maintenance

Trackless machinery will be used for all footwall waste development, and to deliver materials to the required section at each level. Maintenance, servicing and workshop facilities will be provided on surface only, these workshops will provide the necessary resources to maintain the vehicles and provide for weekly and monthly scheduled servicing. Major overhauls and component swap outs will be performed off-site in the near-by town of Swakopmund.

Designated surface areas will be provided for trackless machinery to ensure the daily maintenance and operating checks are complied with. In addition, these designated areas will be used for fuelling, lubrication and washing of the machines on a daily basis.

No underground workshops have been planned in this study.

2.4.5 Personnel transportation

Employees are either transported to the mine via a bus service or use their own transport. Underground employees proceed to the change house, change into their personal protective equipment (PPE), collect underground lamps and self-rescue pack and report to the shaft bank for clocking in and transport underground. Underground employees will be transported to and from their specific underground working areas by LDV’s ergonomically equipped with appropriate level of Roll Over Protection Systems (ROPS) and Fall On Protection Systems (FOPS) to ensure employees safety whilst being transported in the LDV’s. Persons may not travel by foot in the declines between specified “up-down” times due to haulage truck movement within the declines unless “escape cubby’s” have been provided at specific and regular intervals as prescribed in the mine standards.

2.4.6 Excavation dimensions

Table 2.10 provides the dimensions of various excavations planned to be developed for the Project, as well as their orientation to the horizontal.
## Table 2.10  Access infrastructure and standard stoping dimensions

<table>
<thead>
<tr>
<th>Excavation</th>
<th>Width (m)</th>
<th>Height (m)</th>
<th>Area (m²)</th>
<th>Density (t/m²)</th>
<th>Tonnes/Metre</th>
<th>Dip</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Off-reef development</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ramp (North Decline)</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:6 - 1:7</td>
</tr>
<tr>
<td>Rehab Ramp (South decline)</td>
<td>3.2</td>
<td>2.5</td>
<td>8.0</td>
<td>2.6</td>
<td>20.8</td>
<td>1:6 - 1:7</td>
</tr>
<tr>
<td>Ramp Access</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:6 - 1:7</td>
</tr>
<tr>
<td>FW Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:200</td>
</tr>
<tr>
<td>Loading Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:200</td>
</tr>
<tr>
<td>Muck Bay</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:200</td>
</tr>
<tr>
<td>Refuge bay</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:200</td>
</tr>
<tr>
<td>Material bay</td>
<td>2.3</td>
<td>2.6</td>
<td>6.0</td>
<td>2.6</td>
<td>15.5</td>
<td>1:200</td>
</tr>
<tr>
<td>Travelling way</td>
<td>1.8</td>
<td>2.1</td>
<td>3.8</td>
<td>2.6</td>
<td>9.8</td>
<td>34.0°</td>
</tr>
<tr>
<td>Return Air Ways</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>1:200</td>
</tr>
<tr>
<td>Return Air Raise</td>
<td>2.1</td>
<td>3.5</td>
<td>2.6</td>
<td>9.0</td>
<td>45-90°</td>
<td></td>
</tr>
<tr>
<td><strong>On-reef development</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>3.45</td>
<td>35.0</td>
<td>1:200</td>
</tr>
<tr>
<td>Travelling way</td>
<td>1.8</td>
<td>2.1</td>
<td>3.8</td>
<td>3.45</td>
<td>13.0</td>
<td>True dip</td>
</tr>
<tr>
<td>Raise</td>
<td>1.5</td>
<td>2.1</td>
<td>3.2</td>
<td>3.45</td>
<td>10.9</td>
<td>True dip</td>
</tr>
</tbody>
</table>

### 2.4.7  Scheduling rates

The development advance rates shown in Table 2.11 were requested by NRR. Snowden regards these as aggressive but achievable under ideal conditions. These advance rates were used in the Mine2_4D modelling process to determine the build-up to steady state production and beyond.
Table 2.11 Development advance rates

<table>
<thead>
<tr>
<th>Excavation</th>
<th>Width (m)</th>
<th>Height (m)</th>
<th>Area (m²)</th>
<th>Density (t/m²)</th>
<th>Tonnes/Metre</th>
<th>Inst. Rate (m/month)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Off-reef development</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ramp (North Decline)</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Rehab Ramp (South decline)</td>
<td>3.2</td>
<td>2.5</td>
<td>8.0</td>
<td>2.6</td>
<td>20.8</td>
<td>400</td>
</tr>
<tr>
<td>Ramp Access</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>FW Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Loading Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Muck Bay</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Refuge Bay</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Material bay</td>
<td>2.3</td>
<td>2.6</td>
<td>6.0</td>
<td>2.6</td>
<td>15.5</td>
<td>45</td>
</tr>
<tr>
<td>Travelling way</td>
<td>1.8</td>
<td>2.1</td>
<td>3.8</td>
<td>2.6</td>
<td>9.8</td>
<td>35</td>
</tr>
<tr>
<td>RAW Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>2.6</td>
<td>26.4</td>
<td>45</td>
</tr>
<tr>
<td>Return Air Raise</td>
<td>2.1</td>
<td>3.5</td>
<td>2.6</td>
<td>2.6</td>
<td>9.0</td>
<td></td>
</tr>
<tr>
<td>On-reef development</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Loading Drive</td>
<td>3.9</td>
<td>2.6</td>
<td>10.1</td>
<td>3.45</td>
<td>35.0</td>
<td>45</td>
</tr>
<tr>
<td>Travelling way</td>
<td>1.8</td>
<td>2.1</td>
<td>3.8</td>
<td>3.45</td>
<td>13.0</td>
<td>35</td>
</tr>
<tr>
<td>Raise</td>
<td>1.5</td>
<td>2.1</td>
<td>3.2</td>
<td>3.45</td>
<td>10.9</td>
<td>35</td>
</tr>
</tbody>
</table>

A scheduling template has been applied in detail within Mine2_4D mine planning software. Equivalent scheduling rates and time delays derived from this scheduling template has been applied to the remaining areas.

2.5 Underground staffing requirements

The staffing complement for the underground project is based on the operating characteristics of the mine, i.e. fully mechanised, near surface with access via declines and operating on an 11 shift fortnight i.e. a nominal 22-day month. The labour requirements have been based on work rosters as explained in Section 2.3.7.

A detailed review of all manpower requirements has been made and costed in line with latest remuneration trends. These detailed schedules are available as part of the Human Resources costing section of the Project financial model.

The following organisational charts illustrate the underground mining labour, engineering structures, Finance, human resources and Safety Health Environment and Quality (SHEQ) at full complement.
Figure 2.23 Organisational structure – underground development and production

Source: Snowden, 2014
Figure 2.24 illustrates how the underground labour builds up, as well as the quantum for the mining overhead and engineering related labour complement. The underground related labour complement builds up steadily until it reaches about 242 employees before declining as development phases out.

Figure 2.24  Underground labour profile

Source: Snowden, 2014

2.6 Mining schedule

2.6.1 Scheduling constraints

Several scheduling constraints have been applied in the Mine2_4D model. These include:

- The off reef development may not exceed a system advance of more than 45 m/month.
- The on reef and inclined development may not exceed a system advance of more than 35 m/month.
- The numbers of development crews are limited to three mining crews.

These constraints are included in the mine design to ensure that no unrealistic advance rates are applied to the schedule.

Scheduling template

A scheduling template has been applied in detail to the mining footprint within the Mine2_4D mine planning software. Furthermore, time delays e.g. stoping, equipping, have been applied to both the capital and working cost area.

2.6.2 Mining schedule physicals

Mine development has been assumed to start in Q1 2015. Scheduling determined that a mine will have an estimated life of four (4) years in total.
Figure 2.25 to Figure 2.27 summarises the total material movements and metal grades for the project from underground. Underground production builds up from January 2015 until it reaches full production of 20 ktpm by August 2016 for approximately 22 months. Utilising stockpiles the 20 ktpm processing feed can be maintained for 27 months (excluding commissioning tonnages) and followed by a decline in production due to limited mining inventory tonnages and available face length and mine headings.

**Figure 2.25  Total Project material movements**

![Total Project material movements](source)

Source: Snowden, 2014

**Figure 2.26  Material movement and lead-zinc grade**

![Material movement and lead-zinc grade](source)

Source: Snowden, 2014
Figure 2.27 Material movement and silver grade

Source: Snowden, 2014

Figure 2.28 shows the total development metres achieved for the project. The ramp up for development takes about eight (8) months and drops steadily to steady state until maximum ore production is reached.

Figure 2.28 Total development meters LOM profile

Source: Snowden, 2014

Figure 2.29 shows the total mining inventory movement over the LOM per Mineral Resource category. The Indicated Mineral Resource constitutes approximately 65% of the total mining inventory, while the Inferred Mineral Resource is approximately 35% of the total scheduled mining inventory.
2.6.3 Recovered Mining Inventory

Mineral Resource estimate

This Mineral Resource has been classified in accordance with the guidelines of JORC (2012) (van Lente, 2013). The JORC Code sets out minimum standards, recommendations and guidelines for Public Reporting.

The classified Project in-situ Mineral Resource (2013) is reported above a (Pb + Zn) > 1.00% cut-off grade in Table 2.12.

Table 2.12 In-situ Project classified Mineral Resource estimate

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>*Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
<td>529,000</td>
<td>3.45</td>
<td>2.8</td>
<td>5.4</td>
<td>48.2</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>139,000</td>
<td>3.45</td>
<td>2.0</td>
<td>4.3</td>
<td>42.4</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
<td>253,000</td>
<td>3.45</td>
<td>1.8</td>
<td>7.2</td>
<td>39.0</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>7,000</td>
<td>3.45</td>
<td>2.2</td>
<td>3.5</td>
<td>53.4</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>928,000</td>
<td>3.45</td>
<td>2.4</td>
<td>5.7</td>
<td>44.9</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1,000 t to reflect an estimate

Resource Block model

The Block Model ‘nrr_mod_07apr2014.dm’ was used to convert the Mineral Resource to a mining inventory which was generated in Datamine by Snowden Competent Person (CP), Geology. The block model contained the parent cell size of 4 mE by 4 mN by 4 mRL, with 4 sub-cells in the X, Y and Z directions to fill the narrower mineralisation wireframes. Subcells were used in the block model to provide more accurate volumes of the wireframes.
The model was then imported into the Mine2_4D mine planning software tool and checked to establish its integrity. Furthermore, the model was then depleted to the solid model of the mined out stopes. Of interest were the total tonnages and metal content for each resource category reported by the block model.

**JORC 2012 Ore Reserve**

Snowden could not declare an Ore Reserve at this stage of the study due to the following reasons:

- The Project Resources (2013 Resource Model) are yet to be proved to have sufficient volume to make the project economically viable
- The LOM mine design and schedule includes the Inferred Mineral Resources.

To declare a JORC 2012 Ore Reserve, the following aspects require review and sign-off by a CP:

- Processing
- Economic
- Marketing
- Legal
- Environmental
- Social
- Government/Regulatory framework.

Snowden understands that NRR is dealing with these aspects independently of Snowden’s mining study.

The tailings resource has been excluded from this study as metallurgical testing is currently being conducted at the time of writing this report.

**Mining inventory conversion factors**

Table 2.13 illustrates the conversion factors used to determine the mining inventory and will be discussing below.

**Table 2.13 Mining inventory conversion factors**

<table>
<thead>
<tr>
<th>Area</th>
<th>Parameters</th>
<th>Units</th>
<th>Shrinkage Stoping (North decline)</th>
<th>Shrinkage Stoping (South decline)</th>
<th>Down-dip Stoping</th>
</tr>
</thead>
<tbody>
<tr>
<td>All</td>
<td>Geological loss</td>
<td>%</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>All</td>
<td>Dilution</td>
<td>%</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>Off-reef development</td>
<td>Ore loss</td>
<td>%</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>On-reef development</td>
<td>Ore loss</td>
<td>%</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>Ore loss</td>
<td>%</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Off-reef development</td>
<td>Extraction ratio</td>
<td>%</td>
<td>100</td>
<td>80</td>
<td>90</td>
</tr>
<tr>
<td>All</td>
<td>Mine Call Factor</td>
<td>%</td>
<td>98</td>
<td>98</td>
<td>98</td>
</tr>
</tbody>
</table>
Geological losses

No geological losses were applied to the mining inventory as prescribed by the Snowden CP, Geology.

Dilution

A 5% dilution was applied to the tonnage and grade of the mining inventory due to mining activities based on the South African gold and platinum experience. This value will be adjusted once the mine is in operation, as more data is gathered.

Ore loss

A 1% ore loss was applied to the two mining methods as shown in Table 2.13.

Extraction ratio

Different extraction percentages were applied per mining area based on experience at similar operating mines as shown in Table 2.13.

Mine call factor

A mine call factor of 98% was applied to account for variation between planned grades and actual recovered metal.

2.6.4 Mining inventory categories

Mining inventory estimates are based on the resource block model, modifying factors, mine design and production schedule. The scheduled mining inventory primarily comprises Indicated Mineral Resource and Inferred Mineral Resource. Table 2.14 presents the mining inventory Mineral Resource classifications for tonnages and grades in each resource category. The Indicated Mineral Resource constitutes approximately 65% of the total mining inventory, while the Inferred Resource is approximately 35%.

Table 2.14 Mining inventory categories

<table>
<thead>
<tr>
<th>Class</th>
<th>Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>425,151</td>
<td>3.38</td>
<td>3.00</td>
<td>5.71</td>
<td>51.2</td>
</tr>
<tr>
<td>Inferred</td>
<td>234,372</td>
<td>3.38</td>
<td>1.77</td>
<td>6.46</td>
<td>36.5</td>
</tr>
<tr>
<td>Total Inventory</td>
<td>659,523</td>
<td>3.38</td>
<td>2.56</td>
<td>5.97</td>
<td>45.95</td>
</tr>
</tbody>
</table>

2.6.5 Conclusion and recommendations

The underground study presents a realistic underground mine design with an achievable underground development and production schedule.

Based on the mining inventory determined by Snowden, the Project has a potential mine life of 4 years (including ramp-up to steady-state), producing 659,523 of mining inventory at an average grade of 5.97% Zn, 2.56% Pb and 45.95 g/t Ag.
2.6.6 Risk

Key risks identified in the study include the following:

- Lack of Ore Reserve statement due to unavailability of modifying factors (e.g. processing recovery and costs) at the time when the mining study was undertaken and proof of economic viability of the project.

- Lack of available skilled labour, such as in-stope RDO, poses a risk to meet the targets at required timeline.

- Minimum vehicle clearances are less the minimum clearance recommended by Snowden, however, the selected vehicle clearances are not unlawful in Namibia.

A full risk register is in Appendix C.
3 Geotechnical study

This geotechnical section was completed by Celtis with comment and recommendations from Snowden.

3.1 Geotechnical properties (Celtis)

3.1.1 Hydrogeology

Ground water inflows are expected to be negligible and the Project is located in arid desert. It is not considered that hydrogeology will affect mining stability in any way.

3.1.2 Geotechnical data

The marbles and calc silicate rocks forming the ore bodies and the country rock are competent and have few joints or foliation related discontinuities. Seventeen (17) drill core logs (NLDD number series) and core photos were examined, joint frequency is very low and the rock quality designation (RQD) values average at 84%.

From the borehole data, RQDs, Q system classification and the stability number $N'$, the Rock Mass Ratings (RMR) were obtained as indicated in Table 3.1.

<table>
<thead>
<tr>
<th>Hole number</th>
<th>Q</th>
<th>RMR</th>
<th>$N'$ for roof</th>
<th>$N'$ for sidewall</th>
</tr>
</thead>
<tbody>
<tr>
<td>NBD003</td>
<td>Median</td>
<td>58.67</td>
<td>72.88</td>
<td>234.67</td>
</tr>
<tr>
<td></td>
<td>75 Percentile</td>
<td>122.45</td>
<td>72.97</td>
<td>489.81</td>
</tr>
<tr>
<td></td>
<td>25th Percentile</td>
<td>44.32</td>
<td>67.87</td>
<td>177.28</td>
</tr>
<tr>
<td>NBD016</td>
<td>Median</td>
<td>22.6</td>
<td>82</td>
<td>90.4</td>
</tr>
<tr>
<td></td>
<td>75th Percentile</td>
<td>42.52</td>
<td>82</td>
<td>170.08</td>
</tr>
<tr>
<td></td>
<td>25th Percentile</td>
<td>10.22</td>
<td>77</td>
<td>40.89</td>
</tr>
<tr>
<td>NDB004</td>
<td>Median</td>
<td>47.2</td>
<td>82</td>
<td>188.8</td>
</tr>
<tr>
<td></td>
<td>75th Percentile</td>
<td>242.35</td>
<td>92</td>
<td>969.39</td>
</tr>
<tr>
<td></td>
<td>25th Percentile</td>
<td>30.36</td>
<td>82</td>
<td>121.44</td>
</tr>
<tr>
<td>NDB013</td>
<td>Median</td>
<td>46.88</td>
<td>82</td>
<td>187.52</td>
</tr>
<tr>
<td></td>
<td>75th Percentile</td>
<td>243.2</td>
<td>92</td>
<td>972.8</td>
</tr>
<tr>
<td></td>
<td>25th Percentile</td>
<td>44.16</td>
<td>82</td>
<td>176.64</td>
</tr>
</tbody>
</table>

Source: Celtis 2013

3.1.3 Geotechnical design

The geotechnical design is based on the median rock properties derived from the geotechnical logging rather than the individual rock types, as the interbedded nature of the rock mass and the crosscutting geometries of the ore shoots result in the full range of rock types forming the sidewalls and backs of the stopes.

The Potvin stability graph method was used to design stable panel spans for the stopes. The method which is widely used incorporates the relevant geotechnical information based on a modification of Q, the Modified Stability Number N, which is related to the hydraulic radius as shown in Figure 3.1.
The relationship between N' and the hydraulic radius derived from Potvin indicates that hydraulic radii of greater than 16 were determined. This relationship is stable without support for the hanging-wall, back or roof. This analysis was based on the median N’ values for the poorest hole NDN 16, the overall median values are considerably higher. The stable spans for stoping without support are shown in the Table 3.2.

**Table 3.2 Maximum stable, unsupported spans for the Namib project**

<table>
<thead>
<tr>
<th>Length (m)</th>
<th>Hangingwalls</th>
<th>Backs</th>
<th>Hydraulic Radius</th>
</tr>
</thead>
<tbody>
<tr>
<td>300</td>
<td>30</td>
<td>70</td>
<td>13.6</td>
</tr>
<tr>
<td>200</td>
<td>40</td>
<td>50</td>
<td>16.7</td>
</tr>
<tr>
<td>100</td>
<td>50</td>
<td>40</td>
<td>16.7</td>
</tr>
<tr>
<td>50</td>
<td>100</td>
<td>30</td>
<td>16.7</td>
</tr>
</tbody>
</table>

It is evident that these dimensions exceed the size of the ore shoots which have been mined and also the probable dimensions of the planned stoping.

Table 3.3 shows the dimensions of the stable stopes calculated from the horizontal spans and ore body widths of all the currently targeted ore shoots from the geological model.
### Table 3.3  Dimensions of all targeted ore shoots and stable mining spans calculated HR

<table>
<thead>
<tr>
<th>Shoot</th>
<th>Width (m)</th>
<th>Strike distance (m)</th>
<th>Back Hydraulic Radius</th>
<th>Maximum dip span (Height) of Hangingwall (m)</th>
<th>Hangingwall HR</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>5.87</td>
<td>12.14</td>
<td>2</td>
<td>300</td>
<td>5.8</td>
</tr>
<tr>
<td>2</td>
<td>3.82</td>
<td>39.21</td>
<td>1.7</td>
<td>175</td>
<td>16</td>
</tr>
<tr>
<td>3</td>
<td>6.17</td>
<td>49.53</td>
<td>2.7</td>
<td>90</td>
<td>16</td>
</tr>
<tr>
<td>4</td>
<td>8.67</td>
<td>22.73</td>
<td>3.1</td>
<td>300</td>
<td>10.6</td>
</tr>
<tr>
<td>5</td>
<td>6.84</td>
<td>13.52</td>
<td>2.3</td>
<td>300</td>
<td>6.5</td>
</tr>
<tr>
<td>6</td>
<td>2.59</td>
<td>10.11</td>
<td>1</td>
<td>300</td>
<td>4.9</td>
</tr>
<tr>
<td>7</td>
<td>3.9</td>
<td>18.69</td>
<td>1.6</td>
<td>300</td>
<td>8.8</td>
</tr>
<tr>
<td>8</td>
<td>3.45</td>
<td>20.38</td>
<td>1.5</td>
<td>300</td>
<td>9.5</td>
</tr>
<tr>
<td>9</td>
<td>8.23</td>
<td>91.24</td>
<td>3.8</td>
<td>50</td>
<td>16.1</td>
</tr>
<tr>
<td>10</td>
<td>13.59</td>
<td>14.44</td>
<td>3.5</td>
<td>300</td>
<td>6.9</td>
</tr>
<tr>
<td>11</td>
<td>7.31</td>
<td>30.47</td>
<td>2.9</td>
<td>300</td>
<td>13.8</td>
</tr>
<tr>
<td>12</td>
<td>7.04</td>
<td>31.88</td>
<td>2.9</td>
<td>300</td>
<td>14.4</td>
</tr>
<tr>
<td>13</td>
<td>8.53</td>
<td>11.87</td>
<td>2.5</td>
<td>300</td>
<td>5.7</td>
</tr>
<tr>
<td>14</td>
<td>6.35</td>
<td>23.51</td>
<td>2.5</td>
<td>300</td>
<td>10.9</td>
</tr>
<tr>
<td>15</td>
<td>2.52</td>
<td>15.1</td>
<td>1.1</td>
<td>300</td>
<td>7.2</td>
</tr>
<tr>
<td>16</td>
<td>3.88</td>
<td>23.41</td>
<td>1.7</td>
<td>300</td>
<td>10.9</td>
</tr>
<tr>
<td>17</td>
<td>3.29</td>
<td>9.62</td>
<td>1.2</td>
<td>300</td>
<td>4.7</td>
</tr>
<tr>
<td>18</td>
<td>3.85</td>
<td>10.56</td>
<td>1.4</td>
<td>300</td>
<td>5.1</td>
</tr>
<tr>
<td>Average</td>
<td>5.9</td>
<td>24.9</td>
<td>2.4</td>
<td>300</td>
<td>11.5</td>
</tr>
</tbody>
</table>

Source: Celtis 2013

The average shoot can be mined to a vertical height of 300 m without pillars as indicated Table 3.3. The above calculations are supported by the stability of the existing mining excavations which have stood without major scaling for more than 20 years.

#### 3.1.4 Stress regime

The stress tensor in the area has not been measured but from the mine visit no signs of anomalous stresses were observed.

#### 3.1.5 Seismicity

The area is not seismically active and the planned mining depth and volume will not generate major shear stresses.
3.2 Mining Method (Geotechnical)

3.2.1 Planned mining

It is planned to mine ore shoots lying to the northwest of the Junction mining as well as the downward extension of the historically mined South Mine and Junction ore bodies. The planned mining method is shrinkage and down dip. Access will be from the existing declines and drives on the levels.

Figure 3.2 North-West existing mining

The use of scraper mining or shrinkage would be governed by the dip of the ore shoots. The proportion of each is shown in Table 3.4.

Table 3.4 Mining method ratio according to ore shoot dip

<table>
<thead>
<tr>
<th>Mining Method</th>
<th>Dip</th>
<th>Total Tonnes</th>
<th>Percentage Split</th>
</tr>
</thead>
<tbody>
<tr>
<td>Down dip</td>
<td>&lt;55</td>
<td>408,956</td>
<td>43%</td>
</tr>
<tr>
<td>Shrinkage</td>
<td>&gt;=55</td>
<td>532,055</td>
<td>57%</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>941,012</td>
<td>100%</td>
</tr>
</tbody>
</table>

3.2.2 Crown pillars

When mining is undertaken in the deeper parts of the mine, below the existing mining level, it may be necessary in some ore shoots to leave crown or sill pillars before working below the existing workings.

The use of grouting to consolidate the fill to replace such pillars was considered. The logistics of consolidation grouting of the waste fill in the filled stopes will make this both uncertain and uneconomic.

The sill pillars should be stable at thicknesses of 7 m or greater, based on experience and studies from similar mines with less competent rock masses.

Crown pillars will be considered for ore shoot that are outcropping, these would leave 7 m vertical width pillars below the weathered zone.
3.2.3 Layout

In the existing mine the standoff distances between ramps and level access excavations and the mining is often as small as 2 m without adverse rock conditions. In future, the minimum standoff distance for level development should be 10 m for ramps. Where existing ramps and levels are utilised this requirement may be relaxed on the proviso that these areas are assessed on an individual basis for geotechnical risk and may require significant ground support or, in extreme cases, abandonment.

3.2.4 Backfill

There is no geotechnical requirement for backfill in the envisaged mining as the ore shoots are generally discrete, and not large enough to require the support of backfill due to the rock properties of the ore and country rocks.

However, filling of the existing and future mined out stopes to reduce the surface tailings footprint and waste storage requirement may be considered in future value engineering studies. The design of any tailings backfill will depend on the tailings to be placed and the amount of dewatering.

With the height of the existing stope voids it will be vital to ensure that the fill is sufficiently drained and has the required additives to prevent the possibility of liquefaction and backfill run away. In addition, the bulkhead design and filling scheduling will need to be adequate to control the head at the base of the stopes.

Waste filling of the existing stopes can be achieved by tipping of waste rock into existing stopes.

3.3 Support requirements

3.3.1 Access support

The existing decline and drives are not supported at all and are in very good condition despite their age. When slipping of the decline is commenced, and in future development, spot bolting when required will be the only support. This would typically use 1.8 m support units. The type of units to be used will be chosen to be compatible with the equipment which will be available on the mine. Split set type units or grouted dowels will both be acceptable. There is precedent for not requiring systematic support in mines with similar massive rock masses in Southern Africa.

A provision of one bolt or split set for every 3 linear metres of development has been established.

3.3.2 In stope support

In the shrinkage and down-dip mining methods proposed, where the mine workers will have access, it will be necessary to install a systematic pattern of 1.8 m splitsets or grouted dowels into the exposed hangingwall. This will be to prevent minor rockfalls or falls of loose rock injuring workers.

The use of temporary props or poles in the stoping will be required. For the down-dip stoping method 1.5 m hangingwall bolts will be installed on a systematic 2 m by 2 m systematic pattern. Temporary jacks and barricades will be installed at the start of each shift to prevent any rolling rock or material from injuring persons working at the down-dip face.
3.4 Portal design

The current portals were excavated with short boxcuts until unweathered rock was exposed at depths of about 3 m below surface. The portal was then undercut and decline development was continued. Due to the arid climate and thin soils the existing portals are still stable without support. The future portals should be supported with a 2 m pattern of 1.8 m grouted dowels and the boxcut sidewalls should be supported with a layer of fibre reinforce shotcrete.

3.5 Geotechnical summary review (Snowden)

Snowden has reviewed the NRR Geotechnical report; dated 31 January 2014 provided by Celtis and Snowden's summary comments and recommendations are provided below:

- the issue remains that only 4 geotechnically logged holes cover the deposit, with just 2 covering the northern section, and no geotechnical logging of the ore-shoots; hence there is a significant risk of encountering unexpected conditions in the northern zones
- the key reason for one hole (NLDD16) giving lower Q'/Q values than the other 3 holes is that it is the only hole with realistic Jn values; hence it is appropriate to use the geotech classification values from this hole for design purposes, rather than the other holes
- the stope design process now appropriately uses the length-weighted median values of Q', rather than averages
- no modification has been made to the input parameters for Q' or N' as discussed by Snowden and Celtis; Snowden considers several of these parameters to be on the optimistic side
- Snowden estimated the net changes to the HR values from using what it considers to be appropriate N' parameters as follows:
  - Hanging-walls – HR of 12.5 c.f. 16
  - Backs – HR of 11 c.f. 13.
- reducing the design HR may result in some of the larger shoots requiring a sill pillar at mid-height to control hanging wall stability/dilution – see highlighted shoots on table below
- Snowden concurs that utilising open stope methods is appropriate but stresses the need for adequate support and safety procedures in working places
- backfill is not likely to be required unless adjacent shoots have a thin rib pillar separating them (e.g. less than 10 m width); in such a case waste rockfill can be used to support the rib
- the width of some of the shoots exceeds 5 m, hence if these utilise man-entry methods the stope back will need systematic support (split-sets or grouted dowels; these are far more effective than timber props)
- no estimate has been made of potential overbreak/dilution from stope walls; given the vein-style nature of the deposit, this could impact significantly on ROM grade.

3.6 Recommendations (Snowden)

After review of the geotechnical report provided by Celtis on the 31 January 2014, Snowden recommends the following actions:
that mine planning should continue as planned for the Feasibility Study, making appropriate allowance for in-stope support.

• the mine operator undertakes a geotechnical investigation programme, including mapping of the level drives during development, and logging any new exploration drillcore.

• review conditions and plans before commencing stoping.
4 Ventilation

This section is provided by Redbrooke House Associates (RHA).

4.1 Introduction

The mine is located at about 400 m above sea level. With an initial mine depth of about 200 m, heat and heat stress are not expected to cause any problems that require any action other than appropriate ventilation.

This study provides a review of the ventilation requirements and ventilation arrangements initially for mining down to 200 m below surface. This includes identifying applicable design criteria leading to suitable ventilation standards. The project is an underground mine with shrinkage stoping and a production rate of 20,000 t/month. The effects of an increase in mine depth will also be considered.

4.2 Summary findings

- The design criteria are set out in Appendix B. Allowable gas concentrations are those current in South Africa and generally the same as the American Conference of Governmental Industrial Hygienists (ACGIH) with no reductions in exposure concentration for extended shift lengths.

- In respect of respirable quartz, although a standard of 0.2 mg/m³ has demonstrated to eliminate silicosis, most jurisdictions use a value of 0.1 mg/m³ and this standard is proposed for the Project. The quartz content of the host rock is very low or negligible and a limiting respirable dust concentration of 5.0 mg/m³ is proposed.

- The recommended design diesel exhaust ventilation rate is 0.05 m³/s per kW and this should ensure that ambient gas concentrations are less than one third of the standards. The proposed diesel particulate matter (DPM) standard is the same as that recently introduced in Ontario, Canada at 0.4 mg/m³.

- The indicated mine ventilation rate for a production rate of 20,000 tonnes per month is 125 m³/s. A mine ventilation rate of 150 m³/s was confirmed by an examination of the ventilation standards for different activities. The ventilation rate includes an allocation for the decline haulage using diesel powered trucks.

- Development ventilation, particularly for long declines from surface was specifically examined and the use of 100 m lengths of 915 mm diameter low leakage duct with a 45 kW fan is recommended to control leakage, provide adequate diesel exhaust dilution and reasonable re-entry times after blasting.

- For the short developments the fans required using 760 mm duct are 15 kW. Alternatively, compressed air fans could be used such as the Korfmann DV6. The fan has a compressed air consumption of 0.07 m³/s or about 40% more than a standard rock drill used with a jack leg. Although a separate electrical supply is not required, the fan efficiency is at best about one sixth that of electric powered auxiliary fans.

- Optimum airway sizes were considered for lateral airways (including declines), raise drill holes and long hole raises, with the optimum air velocities being 8.0 m/s, 15.5 m/s and 6.0 m/s respectively. For haulage declines, the optimum velocity is greater than that normally used to control dust and additional dust mitigation measures such as water sprays may be necessary.

- The recommended overall ventilation arrangement is to use the haulage declines and the surface shaft as intake airways. Ventilation air flows both north and south on the sublevels to the ore shoots and back to Junction stope for exhaust to surface.
For intake between surface and the working areas, it is suggested that two declines should be used in parallel with the surface shaft. Haulage decline sizes are assumed to be 3.2 m by 2.5 m for the South decline and 3.9 m by 2.6 m for the North decline; from the ventilation point of view these dimensions do not need to be larger.

The mined out Junction stope is the main return to surface and connected to the stoping areas by existing (collection) drives. The connection through the crown pillar to surface should be enlarged to 18 m² (3.0 m by 6.0 m).

A feature of shrinkage stoping is the variable cross sectional area for airflow caused by blasting breasts/benches followed by draw down of the “swell”. The contractions and expansions created cause shock losses leading to higher pressure losses. Assuming that the velocity is limited to a maximum of 4 m/s (determined by the main fan pressure available), the pressure losses in each contraction and expansion is about 10 Pa.

Detailed ventilation network simulations are not necessary at this stage of the design with 90% to 95% of the pressure losses in the main intake and exhaust airways. For the recommended ventilation arrangement, the maximum main fan pressures are about 550 Pa and the total main fan power required is about 120 kW.

With a limited mine life of less than 10 years, a relative low pressure system and reasonable power costs, the main fans do not have to be designed for long life and high efficiency. The duty is suitable for axial flow fans and this can be used to reduce overall costs by selecting a much simpler inlet box arrangement.

The estimated cost of the main fan installation on surface is about USD310,000. The operating fan power and maintenance costs would be approximately USD150,000 per year.

Auxiliary fans are required for both long and short development. The overall auxiliary fan power is estimated to be less than 150 kW with a capital cost of USD75,000. The total annual operating cost (power, maintenance and repair) is also estimated to be USD150,000.
5 Engineering design

5.1 Civil design

Mining activities are supported by the plant infrastructure as detailed in the Tenova Report. The proposed configuration items required for supporting mining activities are described below.

All required civil designs will be done in terms of the engineering design criteria accepted by NNR. All required specifications will be in accordance with SANS specifications as amended. The following table identifies the SANS specification that are applicable for civil design.

Table 5.1 Specification for civil design

<table>
<thead>
<tr>
<th>Specification</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>SANS 10100</td>
<td>Structural Design of Concrete</td>
</tr>
<tr>
<td>SANS 10160</td>
<td>General Procedures and Loadings to be Adopted for the Design of Buildings</td>
</tr>
<tr>
<td>SANS 10161</td>
<td>The Design of Foundations for Buildings</td>
</tr>
<tr>
<td>SANS 10162</td>
<td>The Structural Use of Steel</td>
</tr>
<tr>
<td>SANS 1200</td>
<td>Standardised Specifications for Civil Engineering Construction</td>
</tr>
<tr>
<td>BS 8007</td>
<td>Design of Concrete Structures for Retaining Aqueous Liquids</td>
</tr>
<tr>
<td>N.B.R.</td>
<td>The National Building Regulations of 1985 as amended</td>
</tr>
</tbody>
</table>

Source: Snowden (2014)

The distributed load bearing capacity of the concrete, where required, will be civil designs based on information gathered from the geotechnical foundation information from the various areas on the mine property. Civil works for the following structures will be required:

- Mine maintenance building
- Lamp room
- Medical facility
- Mine offices
- Compressor station
- Water storage, pump station and pipeline foundations
- Sewage plant
- Lighting bollards
- Ventilation exhaust area
- Footwalls, sidewalks and ramps for the decline, haulages, travelling ways, access ways, underground water storage areas and refuge bays.
5.2 Materials

5.2.1 Concrete

For ease of application, all concrete applied underground is to be supplied in premixed bags. Appropriate concrete test procedures will be required to ensure that the dynamic weight of the trackless vehicles or other structures does not require the concrete to be re-laid. Concrete mix bags will be transported on material cars down the material decline to the areas where they will be required.

Reinforced concrete will be mixed in accordance with South African Bureau of Standards Code of Practice SANS 10100 - The Structural Use of Concrete - Part 1 - Design (The purpose of this specification is to specify certain details and features not fully covered, and to amplify and add to the requirements of the relevant paragraphs in SANS 10100).

In addition to SANS 10100, the following Codes of Practice shall be consulted as required:

- BS 8110: General Reinforced Concrete Structures and Foundations
- BS 8007: Reinforced Concrete Structures for Retaining Water and Other Aqueous Liquids.

Cube testing is required in all concrete rated 25 MPa and above. The following minimum cube strengths are required after a period of 28 days after pouring:

- reinforced concrete in dam walls, plinths and miscellaneous footings, ground floor slabs, earth retaining walls, walls and pits: 25 MPa
- reinforced concrete in suspended footwall slabs and beams, columns, silos and bins, water retaining structures: 30 MPa
- blinding concrete and mass concrete: 15 MPa.

Curing strength after 24 hours will be tested only if deemed necessary.

5.2.2 Reinforcement

Reinforcing will be supplied as follows:

- high tensile deformed reinforcing steel shall comply with SANS 920 with a minimum yield stress of 450 MPa
- mild steel reinforcing steel shall comply with SANS 920 with a minimum yield stress of 250 MPa
- welded steel mesh fabric shall comply with SANS 1027 with a minimum tensile strength of 485 MPa.

5.3 Mechanical design and engineering

All mechanical designs and construction are also subject to applicable SANS specifications. The mechanical design philosophy for this Project is to utilise the components of the current infrastructure in good condition and to ensure the easy and safe establishment and construction of all new installations and equipping.

The majority of the mechanical installations pertain to the services to support the mining operation and associated equipment.
5.4 Corrosion protection

All permanently installed structures should be painted with a protective coating or galvanised when installed in areas with wet areas or areas of high moisture. It is advisable that pipelines be painted according to a colour coding scheme to ensure applicable maintenance is achieved.

Specific equipment and items that are easily worn are to be inspected on a regular basis to determine if the protective coating requires attention. These include pump and pump station steelwork due to water/humidity exposure in these areas.

5.5 Electrical design and engineering

The complete electrical system will be designed for cost-effectiveness, continuous and reliable service, safety of personnel and equipment, ease of maintenance and operation, minimum power losses, protection of mechanical equipment, inter-changeability of equipment and to provide for future additions. Power supply will need to be managed to ensure that both surface and underground requirements are catered for.

System protective devices shall be selected and coordinated to ensure that the interrupter nearest the point of fault or high overload will open first and minimise stem disturbance.

The latest applicable editions of the following international specifications, standards and codes must be mentioned for the design, construction and testing of equipment. This list is a preliminary list only and as such is not complete:

- NRS 004: Mini-substations
- NRS 005: Distribution transformers
- NRS 003-1: Metal clad switchgear for rated A.C. voltages above 1kV and up to and including 24kV
- IEC 60947: Low Voltage switchgear and control gear
- SANS 10114: Interior Lighting
- SANS 10139: Fire detection and alarm systems for buildings

5.5.1 Power supply, distribution and consumption

The end user voltages will be 11kV, 525V for operations, 400/230V for lighting and small power, and 110V AC for control circuits.

Erongo power supply

The Erongo power supply and the electrical infrastructure to supply the Project will have to be reviewed at the execution stage, to ensure the contractual agreements and battery limits between Erongo and the project are finalised. The main power supply to the NNR complex is 11kV.

5.5.2 Low Voltage distribution

The various sub-stations on site will supply 550V and 400V distribution transformers and mini substations with 11kV. The 550V and 400V fault levels must not exceed 15kA for distribution transformers and mini sub-stations.
11kV/550V will be distributed at 3-phase to distribution boards, motor control centres (MCCs), mining gully-rigs and lighting transformers. All transformer 550V star-connections will be earthed via a resistor to limit earth fault current to 5A.

11kV/400V will be distributed at 3-phase 230V and neutral to surface distribution and lighting distribution boards. All transformer 400V star connections will be solidly earthed.

550V/190V will be distributed at 3-phase 110V and neutral from underground lighting transformers. All transformer 190V star connections will be solidly earthed.

110V will be distributed at 2-phase 55V with the centre tap earthed in motor control centres for control.

5.5.3 Switchgear

Medium Voltage

Medium voltage switchgear will be insulated. The surface consumer substation switchgear will be a double bus arrangement. The ventilation, refrigeration and underground level and pump station substations will be single bus arrangement.

Switchgear and Low Voltage

Low voltage switchgear will distribute power at 400V and 550V. All 550V and 400V distribution switchgear will be rated at 15kA.

The main pump station located at the lowest level of operation will be supplied from the Level 8 station switchboard.

Underground distribution

11kV will be stepped down to 550V, which in turn will be stepped down to 190V for lighting, and 110V for control circuits. 110V will be stepped down to lower voltages.

Protection and earthing

The 630 kVA station transformers are to be protected by means of standard over-current and earth-fault protection relays, Bucholtz, oil and winding temperature trips, oil surge trip and restricted earth fault trip. The restricted earth-fault will operate via a current transformer.

Emergency power supply

No dedicated emergency power supply system has been allowed for in this study except in the refuge bays.

5.5.4 Transformers

Transformers and mini substations

All transformers shall be capable of operating continuously and without adverse effects or overheating under all specified conditions of operating, including variations in system of ±10% in voltage, frequency or both.
Distribution transformers and mini sub-stations for both surface and underground (as required), will be oil insulated, double wound, three phase units and will be equipped with an off-load tap change facility giving five equal steps of 2.5% i.e. ±5% of the rated full load primary voltage. The cooling method for all distribution transformers shall be with oil.

Surface lighting transformers shall be oil insulated, double wound, three phase units and will be equipped with an off-load tap change facility giving five equal steps of 2.5% i.e. ±5% of the rated full load primary voltage. Surface lighting transformers shall be cooled with oil.

Underground lighting transformers, as required, shall be dry type, double wound, three phase units and will be equipped with a lighting distribution panel and off-load tap change facility giving five equal steps of 2.5% i.e. ±5% of the rated full load primary voltage. Underground lighting transformers shall be air cooled.

5.5.5 Cables and cable support

Low halogen PVC sheeted XLPE insulated steel wire armoured copper cable with internal earth will be used for surface and underground reticulation.

Low halogen PVC sheeted paper insulated steel wire armoured copper cable will be used for reticulation in the South vertical shaft. Low halogen PVC cabling must comply with SANS 1507 (1-6).

For medium voltage distribution, XLPESWA copper cable with internal earth will be used for surface reticulation. PILCSWASWA copper cable will be used for reticulation in the South vertical shaft. XLPESWA copper cable with internal earth will be used for reticulation on the respective levels and in North decline.

Galvanised cable racking will be used underground only for sub-stations and pump stations. Galvanised straining wire suspended from eyebolts will serve as dedicated cable routes. Galvanised straining wire supported by eyebolts, and suspended from chains when required, will serve as dedicated cable routes.

Galvanised cable racking, supported on concrete plinths with galvanised posts, will form dedicated cable routes on the surface. These routes will follow general service routes, where possible.

5.5.6 Underground lighting

Small power reticulation for lighting will be powered via the existing infrastructure, with a standard 25kVA lighting transformer and distribution board. Underground lighting will operate at 110V.

5.5.7 Fire-fighting, protection and detection

Basic fire-fighting equipment shall be provided for by way of fire extinguishers located conveniently in the sub-stations and power distribution points. Additional extinguishers should be positioned at the level stations, typically near cross-cuts. Potable water can be used when applicable.

A fire detection system for the sub-stations and mining areas has been allowed for.

Airflow meters should be installed at strategic positions, as specified in the ventilation engineering design, to monitor airflow to and from the underground workings, which will assist decision-making during fire emergencies underground.
Fire-protective coatings must be allowed for, for cabling in sub-stations, mini sub-stations, motor control centres (MCC), distribution boards and at the motor drives for pump stations.

5.6 Control and instrumentation

It is not intended to describe the functionality of the equipment and systems in any detail, but rather to clarify the basis of design.

5.6.1 Networks

Networks where required are to be dedicated, upgradeable and of proven, reliable technology to suit each networking application for voice and data, radio, instrument control and access control systems.

Some of the networking systems may be wholly or partly integrated with another and therefore, may share some functionality; for example, radio/telephone.

Programmable Logic Controls (PLCs) will form part of the MCC’s and remote I/O will form part of MCC’s or standalone panels. Major processes will be controlled by dedicated PLC’s in conjunction with remote I/O’s, in the applicable sections. Critical areas, such as pump stations, will have standalone PLC’s.

Sump pumps will be started and stopped under the control of level switches. Manual start facilities will also be provided adjacent to the respective pumps.

UPS requirements for surface and underground operations are critical for communications and critical areas only.
6 Mining infrastructure and utilities

The Project requires infrastructure and services to meet the requirement to support the underground mining operation and mobile equipment used in the processing section. One mine maintenance building is to cater to the maintenance and repair of mobile equipment, and servicing of other underground equipment including rock drills, pumps and other equipment. Consequently, this building will be erected in the vicinity of the North and South mining ingress.

The current scope of the Project includes the refurbishment of surface buildings and the establishment of new structures. Other activities include the connection of utilities.

6.1 Acts and regulations

Equivalent of:

- Minerals (Prospecting and Mining) Act, 1992 (Act No. 33 of 1992) and Regulations
- The Mine Health and Safety Regulations (10th Draft)
- The Occupational Health and Safety Act, Act 78 of 1973 and Regulations
- The Occupational Diseases Mines and Works Act, Act 130 of 1993
- The Atmospheric Pollution Prevention Act, Act 45 of 1965
- The National Environment Management Air Quality Act, Act 39 of 2004
- Petroleum Products and Energy Act, 1990
- Petroleum Products and Energy Amendment Act, 2000
- The Environmental Conservation Act, Act 73 of 1989
- The Hazardous Substance Act, 1974

For site installation, testing, commissioning and training:

- The National Building Regulations and Building Standards Act, Act 103 of 1977
- The Basic Conditions of Employment Act, Act 75 of 1997
- The Workman’s Compensation Act, Act 30 of 1941, as amended
- The Unemployment Act, Act 63 of 1977, as amended
- The Compensation for Occupational Injuries and Diseases Act, Act 130 of 1993
- The Health Act, Act 63 of 1977
6.2 Road access

The Project is accessed from Route B2, the main Swakopmund-Windhoek highway. This is a tarred dual road. The entrance to the Project is along a compacted dirt road. All internal roads will be in a similar condition. Due to the low rainfall experienced in the Project area (see Section 1.4.2), drainage along the surface access ways to various section of the property will not be required.

6.3 Surface layout and buildings

The mining operations will be sustained and supported by a variety of buildings, and other infrastructure to provide specific services. These include:

- Compressor station housing compressors supplying the underground production areas for the hand-held development and production drilling, as well as a constant supply feed to the refuge bays.
  - Excess capacity from the compressor, will be made available for the processing plant.

- Lamp room and crush for production, maintenance and operating staff working underground will cater for the storage of cap laps, rescue packs and measuring and monitoring devices. Maintenance of these items will also take place in this facility.

- Extension of the change house completed for processing plant staff to cater for an additional 250 people working at the mining operation. The change house facilities will be shared by the various disciplines, such as mining, processing and management staff.
  - The change house facilities are to include facilities to cater for female staff and contractors. Contractors and visitors will also make use of this change house whilst on site.

- Offices for mining staff, independent of the processing and main offices. The offices will be established for the management and administration team involved in the Project. Senior management staff will have dedicated offices, whilst other staff members will share office space.
  - The allocation of offices has not been included, since this will be determined by NRR management.

- Mine maintenance building for trackless mobile mining equipment. The suite of trackless equipment and mobile equipment used at the processing plant and overall on site will be serviced and maintained within this structure.

- Medical structure for minor injuries and patient stabilisation. The facility will be equipped to ensure that first aid and primary treatment is possible.

- Power yard to distribute incoming power from the Erongo Regional Electricity Distributor Company

- Main and emergency lighting for the operating areas and ingress into the mine

- Catchment drains and sumps for water disposal

- Parking area for approximately twenty vehicles

- Simple bus shelter for employees.

The services included in the capital development provided for in the estimation of the mining portion are:

- Compressed air
• Mine service water
• Potable water (inclusive of fire water)
• Return/grey mine water.

Infrastructure for service requirements include:

• The compressor station, and the compressed air piping from the compressor station to the North and South mine portals
  - Compressed air is piped down the South vertical shaft and distributed from the shaft column to the stations of the various operating levels. Piping is extended to battery limit, namely fifteen metres along the main haulage. In the operating condition, piping was designed to ensure correct pipe sizing to the furthest production area.
  - Compressed air supply from the compressor station to the processing plant is to be included as back-up volume for processing operations.
• Piping for water from both the potable/fire water and the process storage tanks feeding surface and underground requirements
  - Piping, delivering water and piping for return water, are arranged in the South vertical shaft and to/from the shaft column to the various operating levels, in the same pipe group with the compressed air line. Similarly, piping was sized to ensure correct flow and pressure conditions to the furthest production area.
• Cabling for power from the main MCC at the processing plant to supply electricity for the surface and underground mining operations.
• Refuge bays on all underground levels will be supplied with essential services, thus ensuring the safety requirements are met as from the initiation of the project.

Electrical reticulation will be aligned along pipe racks or routed and elevated on overhead poles.

6.4 Underground layout

For the underground operation relating to the production areas, services for compressed air feed, water reticulation and power supply are required. Service pipes will be sidewall mounted, and therefore, the wall surface is to be prepared for support chains and pipe hangers.

HDPE piping will be used and based on pressure rating specifications. The implications of using steel piping include having cold galvanising and paint to cater for touch-ups after the completion of the installation of the galvanised pipes.

Compressed air will be piped to the development and operating areas from the surface compressed air station.

Service and potable water will be gravity fed to the development and operating areas from the surface water storage tanks located at the processing plant. Pressure reducing stations will ensure that the pressure rating on the pipes is not exceeded. In this instance a ratio of 2:1 will be necessary.

Dewatering and drainage will be initiated with vertical spindle pumps to remove water from the production areas and any excess water. Drains will channel the water away from the working areas back to the level station, where it will be pumped to the surface.
6.4.1 Compressed air supply

The compressed air volumetric rate must maintain the required steady-state production target of 20 ktpm for the Project over the LOM.

The compressed air supply to both North and South mining areas is from a centralised compressor station on surface, supplying compressed air via overland columns to the South vertical shaft. One compressor, designed at 218 m$^3$/min or 4.36 kg/s, will deliver sufficient compressed air to supply the production requirements, as well as the base load of the refuge chambers and introducing 30% leakage in the piping network.

Excess compressed air will be made available to the processing plant, and its volume will be based on reducing leakages on the pipe routes in the mining section. Consequently, since the tonnage to be extracted in the operation remains at 20 ktpm or less, no additional compressor is required in the compressor station. A standby compressor will be sourced from external suppliers as required.

The following air compression reticulation will be allowed for underground:

- Air for the drop raising (Seco S36 drill), automatic rock drills for development (S25 rock drills) automatic rock drills for stoping (S215 muffled rock drills).

In comparison with the drilling air demand, the air demand for the dewatering pumps is negligible. Furthermore, the air requirement for the refuge bays will also be minimal.

During any emergency period, drilling air will not be required. Thus, there will be a dedicated airflow to the refuge bays only.

The design criteria used to determine the air requirements is based on the production schedule in Table 6.1.
Table 6.1  Compressed air requirements

<table>
<thead>
<tr>
<th>Parameters for compressed air</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Monthly required compressed air requirement</td>
<td>0.207 kg/sec/kt/month</td>
</tr>
<tr>
<td>Monthly production days</td>
<td>23 production days/month</td>
</tr>
<tr>
<td>Number and duration of shifts</td>
<td>2 by 10-hour shifts/day</td>
</tr>
<tr>
<td>Duration of drilling per shift</td>
<td>8 hours drilling/shift (16 hours/day)</td>
</tr>
</tbody>
</table>

20 ktpm ore and 3 ktpm of waste for both decline shafts. The total ROM production is 23 ktpm

Source: Snowden, 2014

The mass of air required to produce ore and waste is determined by the equation 20 ktpm * 0.207 kg/s/ktpm = 4.14 kg/s. With a dry air density of 1.2 kg/m³, the air requirement is 3.45 m³/s or 7,300 m³/hr. Allowing 30% for losses due to air leakage along the pipe distribution network and spare capacity, the actual air required for the system is 9,503 m³/hr total. This is the maximum if every drill and pneumatic ancillary is operated simultaneously.

6.4.2  Refuge chambers

The refuge chambers are to be fully equipped, on each level making provision for at least 35 people per chamber. The refuge bays will have drinking water supplied through two taps, a telephone, electric whistle, chemical toilet, benches, rotating amber light, emergency procedures and a “No Smoking” board, first-aid equipment and survivor packs stored in a galvanised steel container.

Detailed design of an integrated escape and rescue strategy is critical, and escape routes must be reviewed regularly as mining advances.

Figure 6.2  Refuge bay equipping
6.4.3 Water consumption

The expected mine service water consumption will be a function of the production requirements and the specification of the development and stoping rock drills. Typical steady-state stope parameters and mining activity inputs were utilised to calculate supply water requirements.

6.4.4 Mine service water

The calculations took into account that Seco S215 rock drills will continuously utilised, to meet the production profile as per Table 6.2. These rock drills and equivalent models consume water at a rate of 0.18 ℓ/s.

Table 6.2 Supply water parameters for production

<table>
<thead>
<tr>
<th>Mining Activity Parameter</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Face length</td>
<td>20 m</td>
</tr>
<tr>
<td>Stopping width (average)</td>
<td>2.1 m</td>
</tr>
<tr>
<td>Average advance per blast</td>
<td>1.0 m</td>
</tr>
<tr>
<td>Relative density of ore</td>
<td>3.45 t/m³</td>
</tr>
<tr>
<td>Face advance per month</td>
<td>15 m</td>
</tr>
<tr>
<td>Number of crews per raise line</td>
<td>2 crews</td>
</tr>
<tr>
<td>Number of crews per side (north, south, etc.)</td>
<td>3 crews</td>
</tr>
<tr>
<td>Average number of panels per raise</td>
<td>2 panels</td>
</tr>
<tr>
<td>Number of production panels per raise</td>
<td>2 panels</td>
</tr>
<tr>
<td>Number of rock drills per crew</td>
<td>6 drills</td>
</tr>
<tr>
<td>Tonnage produced by production raise line per day (maximum)</td>
<td>621 tonnes</td>
</tr>
<tr>
<td>Number of days per month</td>
<td>23 days</td>
</tr>
<tr>
<td>Stope production tonnage per month</td>
<td>10,507 tonnes</td>
</tr>
<tr>
<td>Assumed development tonnage (maximum)</td>
<td>1,192 tonnes</td>
</tr>
<tr>
<td>Length of drilling in shift</td>
<td>9.0 hours</td>
</tr>
<tr>
<td>Total quantity of water used per shift</td>
<td>340,200 litres</td>
</tr>
<tr>
<td>Total quantity of water used per shift</td>
<td>340 tonnes</td>
</tr>
<tr>
<td>Total quantity of water used per months</td>
<td>7,825 tonnes</td>
</tr>
<tr>
<td>Consumption fraction</td>
<td>0.15 t/Water/t/Rock</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

6.4.5 Potable water

Potable water is to be supplied underground from the column in the vertical South shaft. The 25 NB potable water line serving for potable water will be extended to reach the furthest production areas mined towards the end of the LOM.

Municipal Provision of water is currently from the supply line of the Swakopmund Municipality and a surface storage tank situated at the shaft. This provides one-third storage capacity of the daily use for the offices, change-houses and underground activities. Underground, drinking points are still available at strategic points down the decline, vertical shaft and in the haulages.
Consumption of potable water underground remains unchanged based on the following criteria:

- Underground use 10 ℓ/person/day
- Design flow 0,5 ℓ/s.

The mine infrastructure must sustain 48-hours potable water holding capacity (minimum 1,000 m³ to cover for feed water for mining operations), thus covering the vulnerability of the water supply system from the Swakopmund Municipality.

6.4.6 Fire-fighting water

The 32 NB service water pipeline will also be used for fire water. Fire hydrants are to be installed down the decline and all tied into the mine water supply line (independent of the potable water line). Hose reels with a 30 m reach will be situated every level station along the decline. The line is to be extended to reach the battery limits in the haulage.

6.4.7 Conventional fire extinguishers

Fire extinguishers will be strategically positioned along the decline, the level stations, at the location of the gully boxes, and along the haulages. Trackless mining equipment vehicles are also required to be permanently equipped with fire extinguishers.

6.4.8 Return water

A settler, return water dam and pumping arrangement at the bottom of the main decline will be established to return water to the surface storage tanks for treatment and eventual recirculation. The return water will include any fissure water that is encountered during the operations. The water:rock ratio for fissure water is 0.001 tonne fissure water:1.0 tonne water.

For the total monthly production of 20 ktpm, including waste, the peak dewatering flow rate is 14.4 m³/hour or 4.2 ℓ/s. As a worst-case scenario, no percentage losses to rock or ventilation have been accounted for. The correct fissure water ratio will be defined when information from the geo-hydrological study becomes available. The average dewatering throughput is calculated at 66% of peak requirement, and is also indicated in Table 6.3.

<table>
<thead>
<tr>
<th>Table 6.3</th>
<th>Dewatering flow rates</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dewatering Parameter</td>
<td>Peak</td>
</tr>
<tr>
<td>Daily flow rate</td>
<td>173.58</td>
</tr>
<tr>
<td>Hourly flow rate</td>
<td>14.47</td>
</tr>
<tr>
<td>Volumetric flow rate</td>
<td>0.0040</td>
</tr>
<tr>
<td>Mass flow rate</td>
<td>4.22</td>
</tr>
<tr>
<td>Fissure water (0.001m³/tonne mined)</td>
<td>12.00</td>
</tr>
<tr>
<td>Dewatering volumetric flow rate</td>
<td>0.0154</td>
</tr>
<tr>
<td>Dewatering mass flow rate</td>
<td>16.22</td>
</tr>
<tr>
<td>Hourly flow rate (total)</td>
<td>14.48</td>
</tr>
<tr>
<td>Hourly flow rate (total)</td>
<td>4.02</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

The dirty water will be pumped from level-to-level through the single-stage centrifugal pumps to surface, thus avoiding the requirements to establish cascade dams and intruding additional pumping systems.
6.4.9 Piping support

Where possible, all level piping in the development areas will be sidewall mounted with chains and support beams.

6.4.10 Power supply and electrical reticulation

The electrical power supply is extended from surface down the South vertical shaft and into the level stations. A splitter will be introduced at each level to ensure power supply to both the northern section and southern section operation and cabling will be extend to the battery limits.

The load is calculated by taking into account the distances and power requirements, as well as the major items and equipment in the Mechanical Equipment List. The electrical layout was also considered to match the capacity available to the future requirements of the operation beyond the battery limits.

6.4.11 Transportation and logistics

People transportation

The shift will travel via available vehicular travel or walk down the decline to access the decline level stations, and after descending onto their allocated working level, will proceed to the working place.

Ore and waste transportation

The mining production and development rock is loaded onto the load haul dumpers and carried back to surface along the main decline. On surface, the material will be stockpiled for processing, whilst waste rock will be taken to the waste dump, or used for construction when necessary.

6.5 Mine maintenance building

The present steel frame building on site will be re-sheeted and used as the maintenance building. The facilities within the mine maintenance building are for basic maintenance purposes and minor servicing of equipment. Major maintenance and servicing of equipment will be conducted by specialists from Swakopmund.

Replacement items for rigging and rope-up equipment such as shackles, slings, lifting equipment, spares, tyres, lubrication, etc. will be stored in lockable sea containers under a prefabricated roof for shelter.

A foreman’s office, an administration room, planning office and library, are included for scheduling of the services of the respective vehicles and required personnel. A tearoom and ablution facilities has also be included in the Tenova scope of work.

A scrap store should be made available to contain the scrap and other materials for disposal. Steel waste, oil and grease and general waste will be separated to reduce environmental risks. Dirty oil recovered from the sumps will then be pumped into drums and sent off-site.
Recommended daily service

In daily service, the vehicle or mobile equipment will be cleaned in the wash bay, as it moves into the workshop and onto the service ramps. The equipment will then be checked by a semi-skilled mechanic, greased and oiled, as required. Minor repair or adjustments can also be undertaken.

In the event of a major repair, the planner will be informed to recall the vehicle, as soon as space becomes available in the workshop and specialist maintenance engineers arrive from Swakopmund.

Finally, the vehicle will be refuelled as it leaves the workshop.

Recommended 50/500 hour service

NRR have advised that the more detailed services of plant will be undertaken by specialist support services from Swakopmund.

In this service interval, a number of components are to be either changed or checked. The mobile equipment will be washed as it enters the workshop and parked in the specified major service bay. A skilled mechanic should perform the required service and repairs. Any repairs that cannot be supported by staff will be provided for by an external contractor. This service usually takes four hours and should be repeated approximately every 30 days or 60 days, for a machine working eight hours per day on average.

The mobile vehicles operate for an approximate total of 240 hours per month. This will require at least one major service per week, i.e., four (4) major services per month of which three will be the 4-hour service and one service is an 8-hour service.

Based on a mobile fleet of nine, 180 hours per month is required, implying one service bay required per one shift per day.

Recommended 1,000 hour service

NRR has advised that the more detailed services of plant will be undertaken by specialist support services from Swakopmund.

This will be completed by an external contractor as required. This is a major service with brake systems being attended to. At 240 hours per month, one unit should become available every four (4) days. This service usually takes four (4) shifts to complete. All other components are checked in the normal manner.

One dedicated service bay per one shift per day is required.

Major repairs and breakdown repairs

This will be completed by an external contractor as required. When a machine breaks down as a result of a major component failure, the component will generally be removed after examination in the workshop and replaced with a reconditioned/new unit.

6.6 Medical facilities

The medical facilities will be designed to support minor medical requirements and basic life support in order to allow for external medical attention via ambulance or otherwise to suitable medical facilities in Swakopmund.
6.7 Security

Due to the remote location of the proposed facility, security is not deemed a significant risk.

In terms of employee control, it is suggested that a basic man-management system in place in order to understand who is on site, and if underground, where they are working. This is not only a safety issue but also to allow for time management determination of payroll.

Stores and equipment will be housed in lockable sea containers.

6.8 Communications

Communications will be limited to a leaky feeder system with emergency two-way handheld radios in the underground environment and telephonic or mobile communications on surface.
7 Financial Analysis

7.1 Methodology and inputs

All dollar values in this section are United States Dollars (USD).

Snowden has consolidated the mining and processing schedules and all costs and applied these to a discounted cashflow model as the basis for the methodology. All results are in USD and all values are real dollars although no particular date is applied to the evaluation; the capital expenditure (Capex) and operating expenditure (Opex) were estimated at early 2014.

The cash-flow model was prepared using data determined during the study and was used to extrapolate for the purposes of this study. This cash-flow model provided post taxation financial information on the Project and a financial analysis with a calculation of the Internal Rate of Return (IRR) and the real Net Present Value of the net cash-flow at a discount rate of 8% (NPV8).

This model does not take into account the cost or source of capital or hedging. Similarly, no provision has been made for salvage or mine closure costs as it is anticipated that the project will continue beyond the present 4 year mine life.

The proposed plant to be constructed at the Project has an annual ore processing throughput of 0.25 Mtpa and will be constructed over 12 months. Ramp up of mining has been provided for over the first pre-production year and processing over the first production year. Tenova provided the Capex for the plant, broken down into various departments which have been summarised in the cashflow model.

The pre-production period has been estimated to be 12 months.

The product will be transported from the plant site to the port at Walvis Bay for loading onto ships for transhipment to smelter customers.

7.2 Inputs and assumptions

7.2.1 Mining inventory

The mining inventory is based on the estimated Mineral Resource model. The tonnes and grade were estimated using reasonable industry costs. The tonnes mined, processed and the processed ore grade has been scheduled on an annual basis which Snowden considers reasonable.

It is emphasised that the mining inventory used in this report is not a Mineral Reserve as it includes an estimate based on Inferred Resources.

The output results of the cash-flow model have been provided on a total and annual basis for the life of the Project.

A summary of the mining inventory is presented in Table 7.1.
Table 7.1 Mining inventory used in technical cash-flow model

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total material mined and processed</td>
<td>Mt</td>
<td>0.660</td>
</tr>
<tr>
<td>Project life</td>
<td>months</td>
<td>59</td>
</tr>
<tr>
<td>Pre-production period</td>
<td>months</td>
<td>12</td>
</tr>
<tr>
<td>Mining life</td>
<td>months</td>
<td>56</td>
</tr>
<tr>
<td>Average head grade</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zn%</td>
<td></td>
<td>5.97</td>
</tr>
<tr>
<td>Pb%</td>
<td></td>
<td>2.56</td>
</tr>
<tr>
<td>Ag g/t</td>
<td></td>
<td>45.95</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

7.2.2 Metallurgical processing

The ore is proposed to be processed by primary crushing, grinding and flotation separation at the Project site to produce zinc (Zn) concentrate and a lead (Pb) concentrate containing silver (Ag) credits. A summary of the metallurgical and processing details is provided in Table 7.2.

Table 7.2 Metallurgical processing assumptions

<table>
<thead>
<tr>
<th>Description</th>
<th>Unit</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pre-production period</td>
<td>months</td>
<td>12</td>
</tr>
<tr>
<td>Plant throughput</td>
<td>Mtpa</td>
<td>0.25</td>
</tr>
<tr>
<td>Ore processed</td>
<td>Mt</td>
<td>0.660</td>
</tr>
<tr>
<td>Process life</td>
<td>years</td>
<td>3.85</td>
</tr>
<tr>
<td>Zn concentrate</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average Zn recovery</td>
<td>%</td>
<td>85</td>
</tr>
<tr>
<td>Average mass pull</td>
<td>%</td>
<td>9.6</td>
</tr>
<tr>
<td>Concentrate produced</td>
<td>kt</td>
<td>63</td>
</tr>
<tr>
<td>Grade Zn</td>
<td>%</td>
<td>53.0</td>
</tr>
<tr>
<td>Recovered Zn</td>
<td>kt</td>
<td>33</td>
</tr>
<tr>
<td>Product moisture</td>
<td>%</td>
<td>9</td>
</tr>
<tr>
<td>Steady state average annual concentrate</td>
<td>kt</td>
<td>22</td>
</tr>
</tbody>
</table>

| Pb concentrate                          |      |       |
| Average Pb recovery                     | %    | 85    |
| Average mass pull                       | %    | 3.1   |
| Concentrate produced                   | kt   | 20    |
| Grade Pb                                | %    | 71    |
| Recovered Pb                            | kt   | 14    |
| Grade Ag                                | g/t  | 945   |
| Recovered Ag                            | kozs | 614   |
| Product moisture                        | %    | 9     |
| Steady state average annual concentrate | kt   | 7     |

Source: Tenova, 2014

7.2.3 Market sales of metals

NRR supplied the metal sales prices and future schedule based on 10 bank consensus reports with the price in year 3 applied in year 4.
Table 7.3  Market sales

<table>
<thead>
<tr>
<th>Production year</th>
<th>1 ($)</th>
<th>2 ($)</th>
<th>3 on ($)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zn ($/t)</td>
<td>2,229</td>
<td>2,391</td>
<td>2,415</td>
</tr>
<tr>
<td>Pb ($/t)</td>
<td>2,320</td>
<td>2,337</td>
<td>2,327</td>
</tr>
<tr>
<td>Ag ($/oz)</td>
<td>20.47</td>
<td>20.57</td>
<td>21.83</td>
</tr>
</tbody>
</table>

Source: NRR, 2014

**Net smelter terms**

An expression of interest for offtake terms was provided by MRI Trading AG on 28 March 2014 and 8 April 2014 and these terms were applied in the cashflow model as follows:

Zinc concentrate:
- Payable for Zn content 85% subject to a minimum deduction of 8 percentage units
- Treatment Charge (TC) based on the applicable benchmark between Korea Zinc and Teck of the year of delivery less $50 per tonne of concentrate ($223 - $50 = $173 applied)
- TC base price of $2,000 per tonne of Zn with an escalator on TC of $0.085 for every dollar above the base price
- TC base price of $1,500 per tonne of Zn with a de-escalator on TC of $0.03 for every dollar below the base price
- No penalties.

Lead concentrate:
- Payable for Pb content 95% subject to a minimum deduction of 3 percentage units
- Payable for Ag content 95% subject to a minimum deduction of 50 grams
- Treatment Charge (TC) based on the applicable benchmark of the year of delivery less $35 per tonne of concentrate ($197.50 - $35 = $162.50 applied)
- No escalators were applied
- A refining charge for Ag is expected in the range of $1.75 to $2.00 per ounce of Ag. The rate applied was $1.875 per ounce of Ag
- No penalties.

Table 7.3  Revenue

<table>
<thead>
<tr>
<th>Product</th>
<th>Total $M</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zn</td>
<td>53.758</td>
</tr>
<tr>
<td>Pb</td>
<td>28.442</td>
</tr>
<tr>
<td>Ag</td>
<td>11.597</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>93.798</strong></td>
</tr>
</tbody>
</table>

Source: Snowden, 2014 (based on forecast data supplied by NRR, 2014)
7.2.4 **Capital costs**

**Pre-production Capex**

Over the pre-production period costs will be expended over 12 months.

The total allocation of these pre-production capital costs is outlined in Table 7.4.

The summation of the items in the tables below may not add to the total due to rounding.

**Table 7.4 Pre-production capital cost estimate**

<table>
<thead>
<tr>
<th>Item</th>
<th>$M</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Process Plant</strong></td>
<td>19.282</td>
</tr>
<tr>
<td>Earthworks</td>
<td>0.731</td>
</tr>
<tr>
<td>Civil Works</td>
<td>0.252</td>
</tr>
<tr>
<td>Buildings (Architectural)</td>
<td>0.459</td>
</tr>
<tr>
<td>Structural Steelwork</td>
<td>0.589</td>
</tr>
<tr>
<td>Platedework and Liners</td>
<td>0.101</td>
</tr>
<tr>
<td>Conveyor Mechanicals (Included in Mechanical Equipment)</td>
<td>-</td>
</tr>
<tr>
<td>Mechanical Equipment</td>
<td>4.286</td>
</tr>
<tr>
<td>Piping and Valves</td>
<td>0.975</td>
</tr>
<tr>
<td>Electrical</td>
<td>1.551</td>
</tr>
<tr>
<td>Instrumentation</td>
<td>0.333</td>
</tr>
<tr>
<td>Preliminary &amp; General (P&amp;G's)</td>
<td>3.407</td>
</tr>
<tr>
<td>Transportation of Equipment</td>
<td>0.569</td>
</tr>
<tr>
<td>Commissioning Spares Only (Critical Spares Excluded)</td>
<td>0.134</td>
</tr>
<tr>
<td>First fill of Steel Balls &amp; Reagents</td>
<td>0.113</td>
</tr>
<tr>
<td>First fill of Lubricants</td>
<td>0.070</td>
</tr>
<tr>
<td>Vendor assist during Constr &amp; Comm</td>
<td>0.244</td>
</tr>
<tr>
<td>EPCM</td>
<td>2.687</td>
</tr>
<tr>
<td><strong>OTHER COSTS</strong></td>
<td>0.625</td>
</tr>
<tr>
<td>Owner's costs (Tailings Storage Facility)</td>
<td>0.103</td>
</tr>
<tr>
<td>Contingency @ 12% excluding Owner's Costs</td>
<td>2.055</td>
</tr>
<tr>
<td><strong>Ventilation</strong></td>
<td>0.385</td>
</tr>
<tr>
<td>Main fan and installation</td>
<td>0.310</td>
</tr>
<tr>
<td>Auxiliary fans</td>
<td>0.075</td>
</tr>
<tr>
<td><strong>Mining Costs</strong></td>
<td>3.998</td>
</tr>
<tr>
<td>Rehab Development</td>
<td>0.068</td>
</tr>
<tr>
<td>Waste Development</td>
<td>0.874</td>
</tr>
<tr>
<td>Ore Development</td>
<td>0.117</td>
</tr>
<tr>
<td>Stoping</td>
<td>0.118</td>
</tr>
<tr>
<td>Logistics</td>
<td>0.042</td>
</tr>
<tr>
<td>Rock Transportation</td>
<td>0.298</td>
</tr>
<tr>
<td>Power</td>
<td>0.483</td>
</tr>
<tr>
<td>Water</td>
<td>0.010</td>
</tr>
</tbody>
</table>
Due to the short life of the project, infrastructure has been kept to a minimum as major plant and equipment servicing as well as maintenance being undertaken from off-site specialists in Swakopmund.

**Production capital cost estimate**

The production capital estimate is allocated to mining infrastructure as the mining Opex contains all of the development costs for the mine for the first 3 years. As the processing life is only 4 years there has been no allocation for plant sustaining Capex however routine maintenance has been included in the plant Opex.

The allocation of these production capital costs is outlined in Table 7.5.

**Table 7.5 Production capital cost estimate**

<table>
<thead>
<tr>
<th>Description</th>
<th>$M</th>
</tr>
</thead>
<tbody>
<tr>
<td>Underground infrastructure</td>
<td>0.031</td>
</tr>
<tr>
<td>Compressor infrastructure</td>
<td>0.010</td>
</tr>
<tr>
<td>Potable water infrastructure</td>
<td>0.006</td>
</tr>
<tr>
<td>Mine dewater</td>
<td>0.004</td>
</tr>
<tr>
<td><strong>Total production per year</strong></td>
<td><strong>0.051</strong></td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

**7.2.5 Operating costs**

Operating costs were estimated per tonne of material and total cost for the items shown in Table 7.6.
Table 7.6  Operating costs LOM

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>M Op</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total Mining Cost / t Ore</td>
<td>$/t</td>
<td>30.07</td>
</tr>
<tr>
<td>Total Mining Cost</td>
<td>$M</td>
<td>19.834</td>
</tr>
<tr>
<td>Processing</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average cost / t processed</td>
<td>$/t</td>
<td>19.02</td>
</tr>
<tr>
<td>Total processing cost</td>
<td>$M</td>
<td>12.544</td>
</tr>
<tr>
<td>Product transport</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total transport cost / ore</td>
<td>$/t</td>
<td>7.99</td>
</tr>
<tr>
<td>Total transport cost / concentrate</td>
<td>$/t</td>
<td>63.20</td>
</tr>
<tr>
<td>Total product transport cost</td>
<td>$M</td>
<td>5.270</td>
</tr>
<tr>
<td>Summary</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total operating / tonne ore</td>
<td>$/t</td>
<td>57.08</td>
</tr>
<tr>
<td>Total operating cost</td>
<td>$M</td>
<td>37.648</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

7.3 Royalties

Royalties are payable to the Namibian government at the rate of 3% of the paid value of the metals as shown in Table 7.7.

Table 7.7  Royalties

<table>
<thead>
<tr>
<th>Product</th>
<th>$M</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zn</td>
<td>1.613</td>
</tr>
<tr>
<td>Pb</td>
<td>0.853</td>
</tr>
<tr>
<td>Ag</td>
<td>0.348</td>
</tr>
<tr>
<td>Total Royalties</td>
<td>2.814</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

7.4 Taxation

Taxation methods were supplied by Grant Thornton accountants and were applied at the rate of 37.5% of the taxable income after allowable deductions. Depreciation was calculated on the basis of a straight line over 3 years and initial expensed deductions of $3,000 M was also provided. The taxation paid is presented in Table 7.8.

Table 7.8  Taxation

<table>
<thead>
<tr>
<th></th>
<th>$M</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total taxation</td>
<td>9.359</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014 (based on information supplied by Grant Thornton)
7.5 **Financial statistics**

The cash-flow model provided the overall project financial statistics after taxation and are presented in Table 7.9.

Table 7.9  **Financial statistics**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>NPV₈%</td>
<td>$M</td>
<td>12.445</td>
</tr>
<tr>
<td>Net cash-flow</td>
<td>$M</td>
<td>18.598</td>
</tr>
<tr>
<td>Revenue</td>
<td>$M</td>
<td>93.798</td>
</tr>
<tr>
<td>Cash outflow</td>
<td>$M</td>
<td>75.199</td>
</tr>
<tr>
<td>Peak equity funding</td>
<td>$M</td>
<td>25.156</td>
</tr>
<tr>
<td>Operating cash-flow</td>
<td>$M</td>
<td>53.335</td>
</tr>
<tr>
<td>Production year payback</td>
<td>year</td>
<td>1.4</td>
</tr>
<tr>
<td>IRR</td>
<td>%</td>
<td>38</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

7.6 **Key performance indicators**

The cash-flow model provided a range of useful Key Performance Indicators (KPI) to measure the financial performance of the Project. The KPIs are presented in Table 7.10.

Table 7.10  **Key performance indicators**

<table>
<thead>
<tr>
<th>Description</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Value of metal</td>
<td>$/t ore</td>
<td>142.22</td>
</tr>
<tr>
<td>Total cost</td>
<td>$/t ore</td>
<td>114.02</td>
</tr>
<tr>
<td>Cash cost</td>
<td>$/t ore</td>
<td>61.35</td>
</tr>
<tr>
<td>Pro-rata total costs</td>
<td>$/t Zn</td>
<td>1,876</td>
</tr>
<tr>
<td>Pro-rata total costs</td>
<td>$/t Pb</td>
<td>1,866</td>
</tr>
<tr>
<td>Pro-rata total costs</td>
<td>$/oz Ag</td>
<td>17.50</td>
</tr>
<tr>
<td>Pro-rata cash costs</td>
<td>$/t Zn</td>
<td>1,009</td>
</tr>
<tr>
<td>Pro-rata cash costs</td>
<td>$/t Pb</td>
<td>1,004</td>
</tr>
<tr>
<td>Pro-rata cash costs</td>
<td>$/oz Ag</td>
<td>9.42</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014

The pro-rata total and cash costs are based on the costs being allocated in the same proportion as the contribution of the revenue from the metal to the total revenue.

7.7 **Breakeven analysis – NPV₈**

The breakeven analysis for metal grade and price is based on holding the other metals at the analysis values while change the metal under consideration until the NPV₈ has a value of $0.000.
<table>
<thead>
<tr>
<th>Description</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Diluted ore grade processed</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zn</td>
<td>%</td>
<td>3.06</td>
</tr>
<tr>
<td>Pb</td>
<td>%</td>
<td>0.28</td>
</tr>
<tr>
<td><strong>Metal price</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zn</td>
<td>$ / tonne</td>
<td>1,464</td>
</tr>
<tr>
<td>Pb</td>
<td>$ / tonne</td>
<td>558</td>
</tr>
</tbody>
</table>

Source: Snowden, 2014
### Mine Development Plan

<table>
<thead>
<tr>
<th></th>
<th>IRR</th>
<th>38%</th>
<th>YEAR</th>
<th>1.0</th>
<th>2.0</th>
<th>3.0</th>
<th>4.0</th>
<th>4.85</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>31-Dec-15</td>
<td>31-Dec-16</td>
<td>31-Dec-17</td>
<td>31-Dec-18</td>
<td>6-Nov-19</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Currency</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
<td>USD</td>
</tr>
<tr>
<td>Grade Zinc t</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zinc t</td>
<td></td>
<td></td>
<td></td>
<td></td>
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**Note:** All figures are in USD unless specified otherwise.
8 References


# Abbreviations and definitions

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Annexure 1: Underground drill programme

(This contribution has been provided by NRR for inclusion into this report)

After the initial programme successfully tested the continuity of mineralisation below the old mine workings, it was decided to commence the underground diamond resource drilling campaign, involving approximately 5800m of drilling below the South Mine, 2488m below the North Mine from 5 level exploration development tunnel and a total of 1350m of RC surface drilling.

A total of 1045m have also been planned from 1 Level in the North Mine. This is primarily for upgrading the current resource from inferred to an indicated category.

Drilling underneath the South Mine is aimed at extending the current compliant resources and, as a result, is aimed at increasing the mine life. All drilling planned from the North Mine is aimed at converting inferred resources to indicated resources.

To control and secure its schedules, NRR has purchased a Kempe pneumatic U3-9B diamond drill, which is capable of drilling holes to a maximum length of 300m.

A Kempe specialist was on site for two months to train NRR staff as drillers, as part of the process.

The company also mobilised a second diamond drill, an Atlas Copco 262, to expedite the drilling. This drill is a larger and more powerful diamond drill and, as a result, some excavations are required to prepare its drill pads.

Construction of a 300m exploration drive on 5 Level underneath the North Mine has commenced and is ongoing. This is to create drilling platforms. Drilling from this level will convert inferred resources to indicated resources in a short term. The tunnel is located in the footwall of the mineralisation and hence will also be used for mining purposes in the future.
Annexure 2: Future work plan 2014-2016

(This contribution has been provided by NRR for inclusion into this report)

Work in future will focus on:

- Completion of the Feasibility Studies by Snowden Mining Consultants Ltd and Tenova Bateman and submission of a Mining License Application.
- Drilling to continue on the lower levels of the mine to increase the current resource base.
- Drilling to continue on the Northern ore body of the mine to convert the inferred resources to indicated.
- Regional exploration and drilling, largely following up on VTEM targets and targets generated by CSA. The regional work is mainly aimed at sterilising the remainder of the EPL in preparation of relinquishment of most of the area. Regional exploration will also include systematic geochemical surface sampling
- A 3D model of the host marble horizon through the mine area to be built, to better control and understand the complexity especially in the junction and the central zones. These will allow current drill planning to be optimised.
- Development of the 300m drive on 5 Level underneath the North Mine to continue. This will create suitable drill positions for infill drilling, to convert inferred resources to indicated resources. This will also create a platform for mining and further exploration drilling below the North Mine.

Table 1 shows the planned expenditure for the work programs during the renewal period of the licence in 2014-2016.

Table 1 – Planned Exploration Expenditure on EPL2902; April 2014 – April 2016.

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Appendix A  NRR In-Situ Namib Lead Zinc Project
Resource Estimate
North River Resources PLC
Namib Lead-Zinc Project
Project No. L00569
NRR In-Situ Namib Lead Zinc Project Resource Estimate
December 2013
This report has been prepared by Snowden Mining Industry Consultants ('Snowden') on behalf of North River Resources PLC.

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Prepared By Belinda van Lente
PhD (Geology), Pri. Sci. Nat.
Senior Consultant

Reviewed By Simon Dominy
FAusIMM (CP), FGS(CGeol), FAIG(RPGeo)
Executive Consultant

Issued by: London Office
Doc Ref: 131220_F_JR009-12-2013_NRR In-Situ Namib Lead Zinc Project Resource Estimate.docx

Last Edited: 25/06/2014 3:05:00 PM

Number of copies
Snowden: 2
North River Resources PLC: 2
# Executive Summary

The Namib Lead-Zinc Project is located in the Damara Belt of Namibia, which is one of the world's most significant lead-zinc provinces. The project area is characterized by a succession of sedimentary rocks, intrusions of igneous rocks, and episodic mineralization events. The project aims to define a resource estimate that can support the development of a mineable deposit.

## Introduction

The Namib Lead-Zinc Project is a joint venture between North River Resources PLC and a local Namibian company. The project is situated within the Damara Belt, a region renowned for its lead-zinc deposits. This section provides an overview of the project, including its location, geology, exploration history, and development plans.

### Site Visit

A site visit was conducted in December 2013 to assess the project area and gather field data. The visit included a detailed inspection of the project site, including mineral occurrences, geological features, and accessibility.

### Project Location and Exploitation Area

The project area is located within the Damara Belt, specifically within the Walvis Bay Sub-basin. The area covers approximately 100 km² and is accessible by road and air.

### Exploration and Mining History

Historical exploration in the project area has identified numerous mineralized zones, primarily lead-zinc veins hosted within sedimentary rocks. Previous mining operations, primarily underground, have produced significant amounts of lead-zinc ore.

## Geology

### Regional Geology

The geology of the project area can be divided into several major stratigraphic units:

- **The Palaeoproterozoic rocks**: These are the oldest rocks in the project area and consist of metamorphosed sediments and volcanic rocks.
- **The Mesoproterozoic rocks**: These rocks are younger than the Palaeoproterozoic and include a sequence of sedimentary and volcanic rocks.
- **The Damara Sequence and associated rocks (1 000 – 500 Ma)**: These rocks are characterized by a sequence of sedimentary and volcanic rocks with banded iron formations.
- **Karoo and Cretaceous Rocks (500 – 50 Ma)**: These rocks consist of volcanic and sedimentary deposits that are typically rich in lead-zinc mineralization.
- **Kalahari Sequence and related rocks (50 Ma to present)**: These rocks are composed of sedimentary deposits that have been uplifted and deformed.

### Local Geology

The local geology of the project area is characterized by:

- **Mineralisation**: The main mineralisation is hosted within the Damara Sequence, particularly in the Damara Iron Formation.
- **Deformation**: The project area has been subjected to significant tectonic activity, resulting in a complex structural framework.
- **Structure**: The geology is characterized by a series of folds and fault systems.
- **Down Hole Transient Electromagnetic Study (DHTEM)**: This study was conducted to map the subsurface electromagnetic properties and identify potential mineralized zones.

## Drilling Campaigns

A series of drilling campaigns were conducted to define the resource base. The campaigns targeted known mineralized zones and adjacent potential areas.

## The Resource Database

### Assay

The database contains detailed assay results from all drilling campaigns, including grades and weights of lead and zinc.

### Quality Assurance and Quality Control (QA/QC)

- **Blanks**: The database includes a significant number of blanks to ensure the accuracy of the assay data.
- **Duplicates**: Duplicate samples were taken to verify the accuracy of the assay results.
- **Check Assays**: The database includes check assays to monitor the quality of analysis.

### Bulk Density

The bulk density of the ore samples is critical for calculating the volume and mass of the resource.

### Database Integrity

The integrity of the database is maintained through strict quality control measures.

## Topography Validation

The topography of the project area was validated using aerial photography and ground survey data to ensure the accuracy of the deposit model.

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1 Executive Summary

North River Resources (NRR) engaged Snowden Mining Industry Consultants Ltd (Snowden) to undertake a study on the Namib Lead Zinc Project (NLZP), on its behalf. Dr Simon Dominy is an Executive Consultant in Applied Geosciences at Snowden and is the appointed Competent Person (CP) for this Resource.

NRR is a sole holder of EPL 2902 issued for the NZLP, located 25 kilometres (km) east-northeast of Swakopmund, Namibia. The project area hosts lead-zinc mineralisation which has been mined historically through underground workings between 1965 and 1992, from the Namib lead mine.

The purpose of this report is to serve as supporting documentation to the 2013 Mineral Resource Estimate for the NLZP in-situ mineralisation, reported in accordance with the JORC Code (2012). The Resource was estimated under the guidance of the CP as defined by the JORC Code.

The NLZP is composed of separate bodies with an overall northwest trend and steeply southwest dipping lead (Pb) and zinc (Zn) mineralisation associated with regional sedimentation and metamorphism. Geological, structural and mineralisation models were created based on historical and current 2D and 3D datasets. Separate mineralised ore lodes were modelled using a 1.00% PbZn (Pb % + Zn %) cut-off. A 1 m composite field was used in a geostatistical study (Variography and Quantitative Kriging Neighbourhood Analysis – QKNA) that enabled Ordinary Kriging (OK), controlled by dynamic anisotropy, to be used as the interpolation method. The results of the variography and the QKNA were utilised to determine the most appropriate search parameters and sample numbers.

The parent block size used for the estimation was 4 mE by 4 mN by 4 mRL, with sub-celling down to 1 mE by 1 mN by 1 mRL. The interpolated block model was validated through visual checks, a comparison of the mean composite and block grades and through the generation of section validation slices.

Previously a Mineral Resource, reported in accordance with the JORC code, was prepared in 2012 by CSA Global Resource Industry Consultants (CSA), for the in-situ NLZP, which yielded the following results for Pb, Zn, Ag (silver) and In (Indium):

Table 1.1 In-situ NLZP Mineral Resource Estimate by CSA (2012) - Undepleted

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>*Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
<th>In (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
<td>82,000</td>
<td>3.45</td>
<td>1.8</td>
<td>6.9</td>
<td>39.7</td>
<td>50.3</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>-</td>
<td>3.45</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
<td>495,000</td>
<td>3.45</td>
<td>2.4</td>
<td>6.6</td>
<td>41.8</td>
<td>31.4</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>111,000</td>
<td>3.45</td>
<td>2.7</td>
<td>5.1</td>
<td>60.9</td>
<td>21.7</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>689,000</td>
<td>3.45</td>
<td>2.4</td>
<td>6.4</td>
<td>44.6</td>
<td>32.1</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1000 t to reflect an estimate
The 2013 NLZP Resource model is a rebuilt from the 2012 NLZP Resource model based on the re-interpretation of the mineralisation, informed by historic drillhole data giving confidence in continuity of mineralisation in the Northern Extension and the incorporation of new drilling and channel sampling results in the South Mine.

Snowden estimated three elements, namely Zn, Pb and Ag using CAE Datamine Studio 3™. The results are as follows:

Table 1.2 In-situ NLZP Mineral Resource Estimate by Snowden (2013) - Undepleted

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>Tonnages</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
<td>529,000</td>
<td>3.45</td>
<td>2.8</td>
<td>5.4</td>
<td>48.2</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>139,000</td>
<td>3.45</td>
<td>2.0</td>
<td>4.3</td>
<td>42.4</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
<td>253,000</td>
<td>3.45</td>
<td>1.8</td>
<td>7.2</td>
<td>39.0</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>7,000</td>
<td>3.45</td>
<td>2.2</td>
<td>3.5</td>
<td>53.4</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>928,000</td>
<td>3.45</td>
<td>2.4</td>
<td>5.7</td>
<td>44.9</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1000 t to reflect an estimate.

The block grade estimates compare well with the composites, as well as between the 2012 and 2013 models. The major increase in tonnage is due to the increase in the modelled Resource volumes in the 2013 rebuilt. The Northern Extension ore lodes were re-interpreted, incorporating structural and geological data, as well as Down Hole Transient Electromagnetic (DHTEM) plates as modelled by Southern Geosciences Consultants (SGC). Additional historical drillholes were used for the interpretation as indication to the continuity of ore lodes. The South Mine area showed an increase in the modelled volume due to additional drillhole and channel sampling data.

Table 1.3 Comparison between the modelled Resource volumes for 2012 and 2013

<table>
<thead>
<tr>
<th>Model</th>
<th>Area</th>
<th>Volumes</th>
<th>Density (t/m³)</th>
<th>Tonnages</th>
</tr>
</thead>
<tbody>
<tr>
<td>2012</td>
<td>Northern Extension</td>
<td>175,000</td>
<td>3.45</td>
<td>603,000</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>34,000</td>
<td>3.45</td>
<td>116,000</td>
</tr>
<tr>
<td>2013</td>
<td>Northern Extension</td>
<td>247,000</td>
<td>3.45</td>
<td>853,000</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>43,000</td>
<td>3.45</td>
<td>149,000</td>
</tr>
</tbody>
</table>

*Volumes and Tonnages have been rounded to the nearest 1000.

Grade-tonnage curves comparing PbZn% between the 2012 and the 2013 undepleted Resource models for the NLZP in-situ deposit, all classifications, are shown in Figure 1.1 and Figure 1.2 below.
Figure 1.1 PbZn Grade-tonnage curves (Tonnes above cut-off vs. Cut-off grade) for the in-situ NLZP deposit [Blue – 2012; Red – 2013]
Figure 1.2  PbZn Grade-tonnage curves (Grade above cut-off vs. Cut-off grade) for the in-situ NLZP deposit [Blue – 2012; Red – 2013]
2 Introduction

North River Resources (NRR) engaged Snowden Mining Industry Consultants Ltd (Snowden) to undertake a study on the Namib Lead Zinc Project (NLZP), on its behalf. This included:

- Review the existing drillhole and channel sampling data, as well as the data collection procedures in consideration of the guidelines in the JORC code;
- Rebuild the in-situ NLZP Resource, classify the model and report the Mineral Resource Estimate (MRE) in accordance with the JORC Code (2012);
- Set up targets for a drilling program.

In 2012, CSA Global Resource Industry Consultants (CSA) provided NRR with a Mineral Resource, reported in accordance with the JORC code, for the in-situ mineralisation at the NLZP. Since then more work has been undertaken by NRR with the aim of upgrading the available Resource into higher classification categories and to identify additional tonnage.

The objective of this report is to discuss the estimation procedures for the 2013 in-situ Resource for the NLZP, and report the Resource numbers in accordance with the JORC Code (2012). The Resource was estimated under the guidance of the Competent Person (CP) as defined by the JORC Code, Dr Simon Dominy.

Dr Dominy is an Executive Consultant with Snowden and has more than 25 years of experience in the style of mineralisation similar to that at the NLZP. Dr Dominy supervised Dr Belinda van Lente and did not visit the Namib site.

For the 2013 Resource Estimate, Snowden relied upon technical discussions and information provided by NRR personnel that included an Access database, structural information, and documentation on procedures and previous work done on the NLZP.

2.1 Site Visit

A site visit was undertaken by Snowden between the 18th and 20th September 2013. Amongst the team that visited site, was Dr Belinda van Lente who is a Senior Consultant at Snowden.

The team visited the underground workings, where they reviewed the geology and witnessed underground channel sampling. Furthermore, the team visited the sample storage facility where the remainder samples are being kept, including core. The database and storage was also discussed on-site.

Dr Dominy, the Resource CP, did not visit the site. He supervised Dr van Lente.

2.2 Project Location and Exploitation Area

The NLZP is situated 25 km east-northeast of Swakopmund, Namibia (Figure 2.1). NRR currently holds exploration rights. The project is located within the Dorab National Park.
NRR currently holds an exclusive prospecting licence, EPL 2902, to the NLZP, which was renewed for the fourth and last time on the 28th May 2012, and is valid until 17th April 2014 (Figure 2.2).
Figure 2.2 EPL 2902 renewed to NRR on the 28th May 2013

Source: McCracken, 2012
This project covers an area of 45 km² (Figure 2.2) and hosts known lead-zinc mineralisation, which has been historically mined through underground operations. Mining at the Namib lead Mine started in 1965. In 1992, the mine was put on care and maintenance and most of its infrastructure was stripped.

2.3 Exploration and Mining History

This section was adapted from McCracken (2012), unless otherwise specified.

The first exploration undertaken at the NLZP area was between 1932 and 1934, which included the development of three shafts to a depth of 41 m and 10 diamond holes, which intersected ore at 125 m. This was followed by geophysical surveys between 1934 and 1944, with more drilling undertaken in the 1960s.

The Namib lead zinc mine was developed in 1965 and operated until 1992. Access into the mine was primarily through the vertical South shaft along with two declines, namely the Junction ramp and the North ramp. Other minor shafts were developed for ventilation and emergency escape routes. Most of the mining was undertaken from the South and Junction ore bodies, to between 200 m and 215 m depth, while the North orebody was mined to a 30 m depth.

Figure 2.3 Plan view of the underground workings at the Namib lead zinc mine

Source: McCracken, 2012

In the early years of operation, the mine primarily produced lead concentrate, while zinc was sent to the tailings. Zinc started being recovered from 1974. In 1984, the plant was modified for higher zinc recovery, leading to a substantial increase in zinc production.
Production records indicate that from 1986 to 1991, 356,300 t were milled yielding grades of 5.3% Zn and 1.6% Pb to produce 38,121 t of zinc concentrate and 14,142 t of lead concentrate, respectively. The absence of waste rock dumps on surface suggests that all the excavated rock was processed and disposed of as tailings. The mine closed in 1992, due to low metal prices and financial difficulties.

Between 1992 and 1993, Iscor (Namibia) (Pty) Ltd carried out exploration on the project area, under Grant M46/3/1882; this included percussion drilling at a gossan target (GOS504), located west of the N20 orebody.

A Resource estimate for Zn, Pb and Ag was prepared by Berry in 1993, which yielded the following results (Table 2.1). There are no details as to how this Resource was prepared and it was not reported in accordance with the JORC code.

### Table 2.1 Resource Estimate prepared by Berry (1993)

<table>
<thead>
<tr>
<th>Working Area</th>
<th>Levels</th>
<th>Ore Resource (t)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
<th>Confidence/Classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>North</td>
<td>1 – 2</td>
<td>30,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Development drilling/Proven</td>
</tr>
<tr>
<td></td>
<td>2 – 7</td>
<td>50,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Drilling/Probable</td>
</tr>
<tr>
<td></td>
<td>7 – 13</td>
<td>180,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Extrapolated/Possible</td>
</tr>
<tr>
<td>Junction</td>
<td>6.5 – 7</td>
<td>9,400</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Development/Proven</td>
</tr>
<tr>
<td></td>
<td>7 – 10</td>
<td>57,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Development/Extrapolated/Probable</td>
</tr>
<tr>
<td></td>
<td>10 – 13</td>
<td>57,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Extrapolated/ Possible</td>
</tr>
<tr>
<td>South</td>
<td>7 – 8</td>
<td>60,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Development/Proven</td>
</tr>
<tr>
<td></td>
<td>8 – 10</td>
<td>120,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Development/Extrapolated/Probable</td>
</tr>
<tr>
<td></td>
<td>10 – 13</td>
<td>180,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Drilling/ Probable</td>
</tr>
<tr>
<td>N20</td>
<td>1 – 2</td>
<td>56,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Drilling/Probable</td>
</tr>
<tr>
<td></td>
<td>2 – 13</td>
<td>308,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td>Extrapolated/Drilling/ Possible</td>
</tr>
<tr>
<td>Surface Tails</td>
<td></td>
<td>445,500</td>
<td></td>
<td></td>
<td>2</td>
<td>Auger Drilling/Not Classified</td>
</tr>
<tr>
<td>Total Proven</td>
<td></td>
<td>99,400</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td></td>
</tr>
<tr>
<td>Total Probable</td>
<td></td>
<td>563,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td></td>
</tr>
<tr>
<td>Total Possible</td>
<td></td>
<td>545,000</td>
<td>2.25</td>
<td>7</td>
<td>55</td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>1,652,900</td>
<td>1.64</td>
<td>6.65</td>
<td>40.18</td>
<td></td>
</tr>
</tbody>
</table>

After 1992 the mine was abandoned, which lead to surface infrastructure being vandalised or removed. Flooding at the mine to the 6.5 level was also reported.

In the mid-1990s, a group of investors attempted to reclaim the tailings, but failed supposedly due to the inability to suppress pyrrhotite in the process, which led to low recoveries.

In 2001 an EPL was granted to local geologists who primarily assessed the potential for re-processing the tailings. No further work was done on the project area until 2007.
Between April and June 2007, Kalahari Minerals (Kalahari) through its subsidiary Craton Diamond (CD), completed drilling of 17 diamond holes to test the strike and dip continuity of each orebody, at a total length of 3,057 m. Much of the drilling was done on the N20 orebody, which was previously under explored. The drilling results showed that the orebodies had short strikes, of between 20 m and 50 m, but longer down plunges in excess of 200 m.

Between September and December 2008, 108 reverse circulation (RC) holes were drilled in the North and N20 orebodies, at a total length of 15,929 m. Initially, drilling was done at a spacing of 30 m by 40 m but due to short strike lengths infill drilling was done at 15 m by 40 m spacing. The results of this drilling campaign were used by West Africa Gold Exploration (WAGE), a wholly owned subsidiary of Kalahari, to prepare an in-house Resource estimate (Table 2.2). This Resource was not reported in accordance with the JORC code.

Table 2.2 Resource Estimate prepared by WAGE

<table>
<thead>
<tr>
<th>Area</th>
<th>*Tonnnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
<th>In (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>N20 &amp; North</td>
<td>612,000</td>
<td>3.6</td>
<td>2.3</td>
<td>6.4</td>
<td>42.1</td>
<td>31.0</td>
</tr>
</tbody>
</table>

*Tonnages have been rounded to the nearest 1000 t to reflect this as an estimate

Surface and down hole electromagnetic surveys were used to assist in the drilling programmes.

When NRR took over the project in 2009, it refurbished and surveyed all accessible voids using a cavity monitoring system. Subsequently, underground diamond drilling was undertaken by Gunzel Drilling contractors between 25 March 2011 and 08 July 2011, where 17 holes were drilled targeting the Junction orebody. A total length of 1,314 m was drilled. Following the results of this drilling campaign, NRR dewatered the lower mine to enable further exploration. Following dewatering, and access establishment into the lower parts of the mine, surveying, mapping and channel sampling was undertaken. A total of 544 m from 588 channel samples were collected since June 2012, in developments where mineralisation was intercepted. Three additional RC holes were drilled with a total length of 1,152 m.

In 2012, CSA reported a Resource estimate for the in-situ NLZP in accordance with the JORC code, which yielded the following results:
### Table 2.3  Resource Estimate declared by CSA for the NLZP (2012) in accordance with the 2012 JORC code

<table>
<thead>
<tr>
<th>Cut-off</th>
<th>Class</th>
<th>Density (t/m³)</th>
<th>**Tonnes</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
<th>In (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb%+Zn%&gt;1, NSR&gt;$31</td>
<td>INDICATED</td>
<td>3.45</td>
<td>80,000</td>
<td>1.8</td>
<td>7.1</td>
<td>41</td>
<td>51</td>
</tr>
<tr>
<td>Pb%+Zn%&gt;1, NSR&gt;$31</td>
<td>INFERRED (North Lodes)</td>
<td>3.45</td>
<td>477,000</td>
<td>2.5</td>
<td>6.9</td>
<td>44</td>
<td>33</td>
</tr>
<tr>
<td>Pb%+Zn%&gt;1, NSR&gt;$31</td>
<td>INFERRED (South depth extensions)</td>
<td>3.45</td>
<td>111,000</td>
<td>2.9</td>
<td>5.1</td>
<td>62</td>
<td>22</td>
</tr>
<tr>
<td>Total</td>
<td>INFERRED</td>
<td>3.45</td>
<td>588,000</td>
<td>2.6</td>
<td>6.5</td>
<td>47</td>
<td>31</td>
</tr>
</tbody>
</table>

*Resources have been estimated using a mineralised envelope defined by a cut-off of Zn% + Pb% greater than 1%. Individual 10 m levels for each mineralised zone have only been included in the Resource where the average Net Smelter Return (NSR) dollar equivalent grade for that level in each mineralised zone is greater than $31. This is estimated to be equivalent to the treatment cost of mineralised material that has to be mined.

**Tonnages have been rounded to the nearest 1000 t to reflect this as an estimate.
3 Geology

This section has been adapted from McCracken (2012), unless otherwise specified.

3.1 Regional Geology

Namibia’s surface is covered by rocks ranging in age from Palaeoproterozoic to Phanerozoic and mineral occurrences and deposits are found throughout the stratigraphic column.

Figure 3.1 Geology and Mineral Occurrences of Namibia

![Geology and Mineral Occurrences of Namibia](image)

Source: McCracken, 2012

3.1.1 The Palaeoproterozoic rocks

The Palaeoproterozoic rocks range in age between 2.2 and 1.8 Ga, and include the Kunene and Grootfontein Igneous Complexes in the north, the volcanic Orange River Group, Vioolsdrif Granite Suite, the volcano-sedimentary Khoabendus Group and the Rehoboth Sequence in the central part of the country. These rocks occur as metamorphic inliers which are highly deformed.
3.1.2 The Mesoproterozoic rocks

Mesoproterozoic rocks, dated between 1.8 and 1.0 Ga, are mainly from the Namaqua Metamorphic complex (NMC) in the southern part of Namibia, and the Sinclair and Rehoboth Sequences in the central part of the country (Figure 3.1). The Kalahari Copper Belt (KCB) extends in a north-easterly direction from where the Sinclair rocks occur in central Namibia into north-western Botswana, over a distance of 800 km. The style and host of mineralisation in the KCB has broadly been compared to that of the Zambian Copper Belt. The KCB is hosted in discrete structurally preserved volcano sedimentary horizons that are located between the Damara orogenic belt and the northern margin of the Kalahari Craton (Figure 3.2).

Figure 3.2 The KCB extends north-easterly from Namibia to Botswana

Source: McCracken, 2012
3.1.3 **The Damara Sequence and associated rocks (1 000 – 500 Ma)**

The Damara Sequence underlies most of the central and north-western parts of the country. They comprise of carbonates in the north and a variety of metasediments in the south. Due to extensive bedrock exposure, these rocks have been extensively explored. Most of Namibia’s hard rock mining, including the NLZP, are located in the Damara Belt.

3.1.4 **Karoo and Cretaceous Rocks (500 – 50 Ma)**

The volcanic and sedimentary rocks of the Karoo Sequence occur in the Aranos, Huab and Waterberg Basins in the south-west and northern part of Namibia. The dolerite sills and dykes, basalts and several alkaline intrusions are associated with the breakup of Gondwana.

3.1.5 **Kalahari Sequence and related rocks (50 Ma to present)**

This sequence comprise of aeolian and unconsolidated sediments, which cover most of the south-west and eastern part of the country. Gem quality diamonds are found within the Tertiary and Quaternary sediments along marine terraces and offshore the Namibian coast.

### 3.2 Local Geology

Rocks at the NLZP comprise of marginal shelf facies limestone (now deformed to marble) and subordinate metasediments that have been subjected to intense deformation and folding during the Damara Orogeny.

The Damara Sequence was deposited between 770 and 600 Ma. It consists of the Nosib Group at the base, represented by rift-related siliclastic sediments, and alkali volcanics towards the top of the group. The Otavi and Swakop Groups were deposited concurrently over the Nosib Group (Longridge *et al.*, 2012). The Otavi Group, made up of platform carbonates, was deposited in the Northern Platform and Margin zone on the Congo Craton, while the Swakop Group, made up of carbonates, diamictites and pelites, was deposited to the south of that (Longridge *et al.*, 2012). The Kuiseb Formation overlies both the Otavi and the Swakopmud Group (Longridge *et al.*, 2012).

The regional geology, in Sheet 2214 of the Geological Survey of Namibia, shows a large fold that is defined by the Swakop Group, along with the Nosib Group, which rest unconformably on the Abbabis Metamorphic Complex or Basement, to the southeast of the Omaruru lineament.

Within the project area the fold is defined by the Arises River Member of the Karabib Formation (Swakop Group) which comprises of calcitic marbles.

3.2.1 **Mineralisation**

Mineralisation at the NLZP has a general northwest trend. It is believed to be of sedimentary exhalative (SEDEX) or Mississippi Valley Type (MVT) origin, but locally controlled by folding and thrusting. The mineralisation is defined by short strikes and extensive at depth.
Four major ore zones have been defined at the project, namely the North, South, N20 and Junction, with well-developed gossans developing on surface. The orebodies have sharp contacts, with variable orientation due to folding. The mineralisation generally seems to conform to the marble, but can cut across lithologies. Where gossan is developed it is made up ferruginous material with some galena, cerussite and smithsonite. The gossans often extend to a depth of 10 m, while oxidation extends to 16 m.

Fresh mineralisation consists mainly of sphalerite, galena, pyrrhotite and pyrite. The sphalerite is partly iron-rich due to the presence of marmatite (ZnFeS), an iron rich sphalerite containing more than 6% iron which is difficult to separate from pyrite and pyrrhotite. The marmatite level associated with the sphalerite is unknown. The occurrence of marmatite could have an effect on the iron content in the final zinc concentrate (Lund, 2012).

### 3.2.2 Deformation

Original sedimentary layering (S0) is preserved in the bedding and cross bedding of the Khan and Etusis Formations in the Goanikontes area. Planar bedding and pyritic quartzite bands are preserved in the marbles of the Rossing Formation and sporadically in the Swakop Group.

Dominant fold orientations and their associated foliations produced during D1 and D2 deformation events are not obvious in the project area. One model suggests that F2 trended northwest, prior to the F3 interference folding along northeast-trending axes to produce the characteristic domes of the Central Zone. S1 and S2 are represented by migmatitic banding in the Khan and Etusis Formations, but as laminar flow foliations in the Rossing Formation metapelites.

Williams (1989), amongst others, suggests that the grade of metamorphism in the Damara Orogen increases inwards. Studies of the Central Zone by Nash (1971), Jacob (1974) and Downing (1982) show that staurolite precedes sillimanite, which indicates early, very high kinematic/tectonic pressures. S1 biotite, sillimanite and calc-silicate assemblages were produced by increasing temperatures during D1 which came to an end during the D2 partial melting.

D3 deformation produced ductile flow folding in all marbles and calc-silicate dominated units. An S3 gneissosity and schistosity pervasively replaced migmatitic S1 and S2 banding.

Based on crosscutting relationships between dated granites in the area and ductile, mid-crustal shear zones, the peak of Damaran transpressional deformation (D1 and D2) and metamorphism occurred from 600 to 550 Ma (Bowden et al., 1999). This was followed by constructional tectonics, between 542–526 Ma, which was succeeded by the peak of granite plutonism at about 510 Ma (U–Pb date on monazite; Briqueu et al., 1980; Allsop et al., 1983; de Kock and Walraven, 1994; Nex, 1997). This was succeeded by cooling from ca. 505 to 478 Ma (Bowden et al., 1995, 1999; Jacob et al., 2000), possibly extending to ca. 429 ± 17 Ma (Clifford, 1967). Nex et al. (2001) cited evidence for a post-534 ± 7 Ma but pre-508 ± 2 Ma regional Central Zone heating event (M2, possibly not exactly similar to that of Kasch, 1983), which annealed the great majority of fabrics and minerals in sheeted granites, metasediments and mylonites within high-strain zones. M2 was apparently superimposed on the first regional tectonometamorphic event or M1, which has been dated at 571 ± 64 to 534 ±7 Ma (Nex et al., 2001).
The late to post-D3 event, which produced the garnet-cordierite assemblage, reinforces Sawyer's (1981) study that there are two peaks of regional metamorphism. Kasch (1983) also determined two thermal peaks, separated by a thermal trough for the Southern Zone and Southern Marginal Zone of the Damara Orogen. Kasch’s first metamorphic event (M1) was synchronous with D2. This reached an estimated 590°C while his “post-tectonic” M2 peaked at 570°C. The timing of these two events in the Central Zone was considered to be concurrent with events in the Northern and Southern Marginal Zones.

Puhan (1983) determined peak prograde metamorphic conditions for the southern Central zone of 650°C at 3-4 kbar pressure. Hartmann et al. (1983) determined similar conditions (i.e. high temperature, low pressure) from metapelitic assemblages in the area around Swakopmund.

### 3.2.3 Structure

TECT Consulting (TECT) undertook a structure study at the project area and following is a summary of the observations that came out of the study.

- There is a distinct relationship between the trends of folds limbs, NE-SW trending cleavage and shear zones;
- A cone axis plunging at 48° → 187° with a cone angle of 70° including the spread of poles to compositional banding. The orientation of cataclasite and other shears is similar; suggesting intra limb shearing took place;
- Biotite/quartzite banding is oriented more or less vertically and strikes north-south;
- Sub-vertical magnetite–hematite veins strike towards 030° which is different from the orientation of most ore shoots;
- Ore shoot orientations overlap the distribution of fold axis orientations. Although old axes tend to be about 15° shallower than ore shoots, suggesting that mineralisation may not occur ubiquitously in fold hinges. Basson and Tennant (2012) interprets the mineralisation to be controlled by two structural orientations. The first is parallel to the regional fold hinge zone and the second steeper but sub-parallel to the fold axes. This is likely to be an intersection lineation between fanning cleavage and lithological banding (Basson and Tennant, 2012);
- The upper mined out part of the ore body is associated with brittle-ductile deformation, whereas that deeper more sulphide-rich disseminated mineralisation is relatively in-situ fold controlled material;
- Hornblende-tourmaline-biotite pegmatite bodies which pre-existed mineralisation would have had a damming effect on mineralising fluids and exerted some control on the development of fractures and other fluid pathways;
Source: McCracken, 2012

- Inner margins of NE-SW trending folds elsewhere should be investigated as favourable locations for additional mineralisation regionally; and
- Exploration along strike of the N20 mineralisation is suggested.

Surface measurement of major structures and dykes taken by TECT were imported into CAE Datamine Studio 3™, and projected onto the topography. The strike, dip direction and dip measurements of the primary structures were used to calculate and model 3D structural features downwards (Figure 3.4). These structures were considered during the later modelling of the in-situ ore lodes of the NLZP.
3.2.4 Down Hole Transient Electromagnetic Study (DHTEM)

SGC conducted a DHTEM study on the Northern Extension data of the NLZP. Conductive plate models are produced by using Maxwell software to forward model the response of conductors in the earth by approximating the conductors as flat sheets of uniform conductance and calculating the induced magnetic fields that would be produced by these sheets with the given DHTEM survey parameters and layout (Card, 2013).

The resultant DHTEM plates modelled by SGC were used in conjunction with the structure, lithology and grades to model the resource wireframes. As can be seen in Figure 3.5 below, the plates are good indicators of the strike, continuity and possible extents of the ore lodes. It should, however, be noted that even though it is feasible to use DHTEM data to indicate whether extension below a drillhole exists, it is difficult to quantify the length of the extension (Card, 2013).
Figure 3.5 The 3D DHTEM plates (SGC) and Resource wireframes (Snowden) for the Northern Extension of the NLZP
4 Drilling Campaigns

A summary of the exploration drilling completed on the in-situ NLZP that was used in the Resource delineation and evaluation exercise is tabulated below. The cut-off date for use in the current project was 1st October 2013.
Table 4.1  Drill Campaign Summary of the drillholes used in this Resource delineation and estimation of the in-situ NLZP

<table>
<thead>
<tr>
<th>Drill Program</th>
<th>Description</th>
<th>Hole Number</th>
<th>Number of Holes</th>
<th>Type</th>
<th>Avg. Sample Length</th>
<th>Total Meters</th>
</tr>
</thead>
<tbody>
<tr>
<td>Historic</td>
<td>Historic Drilling (1960’s)</td>
<td>N11 to N38; N48; N52 to N75</td>
<td>53</td>
<td>DD</td>
<td>2.4</td>
<td>10,686</td>
</tr>
<tr>
<td></td>
<td>ISCOR Underground Drilling (late 1980’s to early 1990’s)</td>
<td>PNL43 to PNL50; PNL100 to PNL102; PNL104 to PNL108; PNL110 to PNL112</td>
<td>19</td>
<td>PERC</td>
<td></td>
<td>1,060</td>
</tr>
<tr>
<td>GOS504</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Junction</td>
<td>NRR Underground Drilling (2011)</td>
<td>NLDD018 to NLDD033</td>
<td>17</td>
<td>DD</td>
<td>1.0</td>
<td>1,314</td>
</tr>
<tr>
<td></td>
<td>Kalahari Underground Drilling (2008)</td>
<td>NLDD001 to NLDD017</td>
<td>17</td>
<td>DD</td>
<td>0.7</td>
<td>3,057</td>
</tr>
<tr>
<td>Nam_Pb</td>
<td>NRR Underground Drilling (2008)</td>
<td>NLRC001 to NLRC108</td>
<td>108</td>
<td>RC</td>
<td>1.0</td>
<td>15,929</td>
</tr>
<tr>
<td></td>
<td>NRR Underground Drilling (2013)</td>
<td>NLRC109</td>
<td>1</td>
<td>RC</td>
<td>1.0</td>
<td>200</td>
</tr>
<tr>
<td></td>
<td>NRR Underground Drilling (2013)</td>
<td>NLRCDD001 to NLRCDD002</td>
<td>2</td>
<td>RCDD</td>
<td>0.9</td>
<td>952</td>
</tr>
<tr>
<td>Drill Program</td>
<td>Description</td>
<td>Hole Number</td>
<td>Number of Holes</td>
<td>Type</td>
<td>Avg. Sample Length</td>
<td>Total Meters</td>
</tr>
<tr>
<td>----------------</td>
<td>--------------------------------------------------</td>
<td>-------------------------------------------------</td>
<td>-----------------</td>
<td>------</td>
<td>--------------------</td>
<td>--------------</td>
</tr>
<tr>
<td>UG_CH</td>
<td>NRR Underground Channel Sampling (2011 to 2013)</td>
<td>NLZ_CH_100_01 to NLZ_CH_100_39; NLZ_CH_105_01 to NLZ_CH_100_02; NLZ_CH_110_01 to NLZ_CH_110_15; NLZ_CH_115_01 to NLZ_CH_115_06; NLZ_CH_120_01 to NLZ_CH_120_15; NLZ_CH_125_01 to NLZ_CH_125_02; NLZ_CH_130_01 to NLZ_CH_130_04; NLZ_CH_135_01 to NLZ_CH_135_09; NLZ_CH_150_01 to NLZ_CH_150_04; NLZ_CH_170_01 to NLZ_CH_170_12; NLZ_CH_180_01 to NLZ_CH_180_04; NLZ_CH_235_01</td>
<td>113</td>
<td>CH</td>
<td>0.9</td>
<td>544</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>330</td>
<td>1.0</td>
<td>33,742</td>
<td></td>
</tr>
</tbody>
</table>

The collar position of the holes, including the historical holes, were surveyed using a differential global positioning system (DGPS) by African Geomatics using ITRF2000, epoch 2005.0 coordinate system in 2012.
According to the database the following survey methods were employed (Table 4.2) to locate the collar positions.

**Table 4.2 Collar survey information used at the NLZP**

<table>
<thead>
<tr>
<th>Dataset</th>
<th>Collar survey method</th>
<th>Surveyed by</th>
</tr>
</thead>
<tbody>
<tr>
<td>UG_CH</td>
<td>Leica</td>
<td>NRR</td>
</tr>
<tr>
<td>NAM_Pb</td>
<td>DGPS</td>
<td>G.Symons and African Geomatics</td>
</tr>
<tr>
<td>GOS504</td>
<td>DGPS</td>
<td>G.Symons</td>
</tr>
</tbody>
</table>

There are no available sampling procedures for data collected prior to NRR, except for what was observed in the database. The following is applicable to all drilling since 2008. Logging was undertaken per sampling interval. The material was logged with respect to lithology, grain size and colour, according to the Kalahari logging codes.

The drilled material was sampled over 1 m intervals, which were split using a riffle splitter. Samples were collected into sampling bags and labelled. According to McCracken (2012), the Kalahari protocol stated that field duplicates and blanks were collected and inserted at a rate of 5%. Left over samples were stored in plastic bags. A chain of custody list was compiled, which accompanied samples to the laboratory (McCracken, 2012).

The drilling and sampling procedures as used by NRR are shown in Appendices C and D.

Snowden deems the drilling and sampling process undertaken by NRR and Kalahari to be appropriate and in line with industry standards.
5 The Resource Database

NRR uses a Microsoft (MS) database, named NAMIB_Master_Hist_DB_120723, to capture, manage and store their drilling data. The data stored in the database is all drilling data that is relevant for the mine’s specific geological setting and used to determine and classify the ore reserves. The drilling and sampling data sources were audited by the MSA Group (MSA) as reported by McCracken (2012) and recommendations were made to ensure data reliability and conformity to guidelines set in the JORC code.

Data collected up to October 1\textsuperscript{st}, 2013 from exploration drilling and channel sampling was included in the Resource estimate.

5.1 Assay

All holes drilled for the in-situ resource were sampled and analysed only on intersections with mineralisation, and up to 1 m on either side of these intersections. Samples collected between 2008 and 2011 were transported by road from site to the Genalysis Laboratories in Johannesburg, where sample preparation was undertaken. Once the samples were pulverised a representative sample was collected and sent to Genalysis Perth for analysis (McCracken, 2012). These samples were analysed for 14 elements including Pb, Zn and Ag. The analytical methods used to analyse these elements are listed in Table 5.1 below.

Table 5.1 Analytical methods employed at Genalysis Perth

<table>
<thead>
<tr>
<th>Analysed elements</th>
<th>Code</th>
<th>Method</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ag</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>As</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Bi</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Cd</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Co</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Ga</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Ge</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>In</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Sb</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>Sn</td>
<td>A/MS</td>
<td>4 acid digest with grade analysed using ICP-MS</td>
</tr>
<tr>
<td>V</td>
<td>A/OES</td>
<td>4 acid digest with grade analysed using ICP-OES</td>
</tr>
<tr>
<td>Cu</td>
<td>AX/AAS</td>
<td>4 acid digest with an AAS finish</td>
</tr>
<tr>
<td>Pb</td>
<td>AX/AAS</td>
<td>4 acid digest with an AAS finish</td>
</tr>
<tr>
<td>Zn</td>
<td>AX/AAS</td>
<td>4 acid digest with an AAS finish</td>
</tr>
</tbody>
</table>

Note: Four acid digest used hydrofluoric, nitric, perchloric, hydrochloric

Samples collected from drilling and channels by NRR were composited over 1 m intervals over mineralised intersections, extending to 1 m on either side of the mineralisation. These samples were sent to Bureau Veritas in Swakopmund, Namibia for the analysis of Pb, Zn and Ag.
Snowden deems the analytical procedure followed by Kalahari and NRR to be appropriate.

### 5.2 Quality Assurance and Quality Control (QA/QC)

According to the database field duplicates, blanks and certified reference materials (CRMs) or standards were used in all campaigns but some issues were observed when analysing this data.

The purpose of field duplicates in a QA/QC programme is to assess the precision accompanying the sub-sampling process in the field. According to McCracken (2012), the field duplicates were collected as half core or quarter-cut core for DD and through a riffle splitter in the case of RC drilling. Field duplicates were inserted on site, at a rate of 1:20 or 5%.

Blanks in a QA/QC programme are used to measure levels of contamination at the laboratory especially in the preparation process. Pool filter sand was used as blank material by Kalahari (McCracken, 2012). Blank samples were inserted on site, at a rate of 1:20 or 5%.

The Kalahari samples indicate that CRMs were used, but upon further investigation it emerged that these samples were not certified, thus could not be analysed as CRMs. CRMs should preferably be bought from a reputable supplier, like Analytical Solutions Ltd (AMIS), or certified by a laboratory before the material can be used as such.

#### 5.2.1 Blanks

Blanks analyses were completed using what seemed like detection limits in the database. The blanks analyses showed minimal potential for contamination at the laboratory (Figure 5.1 to Figure 5.3) and Snowden deems the results acceptable.
Figure 5.1   Blanks analyses for the Northern Extension samples
Figure 5.2  Blanks analyses for the South Mine Junction samples
5.2.2 Duplicates

A total of 5% coarse reject duplicate samples were collected at the rig to be analysed with the rest of the samples. Duplicate analyses showed 80% of the duplicate data had a hard absolute relative difference (HARD) of 94%, 78% and 95% for Pb, Zn and Ag, respectively (Figure 5.4 to Figure 5.6). All analyses showed good linear correlation, with correlation factors of greater than 98% (Figure 5.7).

Minimal bias was observed between the original and duplicate samples. It is Snowden’s opinion that the sub-sampling method employed in the exploration samples was good and the insertion rate of duplicates is in agreement with industry best practice.
Figure 5.4 Pb Analysis HARD Plot for Coarse Reject Duplicates for the In-situ samples
Figure 5.5  Zn Analysis HARD Plot for Coarse Reject Duplicates for the In-situ samples
Based on the review, Snowden is of the opinion that the sampling procedures for drilling are aligned with industry best practice. The collected samples were accompanied by blanks and coarse reject duplicates, but no independent CRMs were used in the analysis of this data. The analyses of the QC samples showed that the in-situ samples had good precision for the coarse reject duplicates and low potential for contamination at the laboratory for Pb and Ag.

Due to the lack of appropriate CRMs in the analyses of in-situ samples, Snowden recommended NRR to submit 5% of the original samples as check assays to Intertek Genalysis Johannesburg to ascertain the original sample grades.
Figure 5.7  Linear Correlation Plots for Coarse Reject Duplicates for the in-situ samples
5.2.3 Check Assays

Following upon the Snowden recommendation, NRR submitted 5% of the in-situ samples to an Umpire Laboratory. This was done with the aim to ascertain and improve confidence in the available assayed values. The check assays were evenly selected in space and grade distribution (Figure 5.8 and Figure 5.9). Due to the pulps not being present, the check analyses were undertaken on field duplicates. These were split using a riffle splitter, into about 1 kg samples depending on the remaining sample size.

The check samples were analysed for Pb and Zn at Intertek Genalysis Johannesburg, using a four acid digest with grade analysed using ICP-OES. The analysis of Ag was completed in Perth, due to the Johannesburg laboratory not certified for the analytical method used for Ag, which is a four acid digestion with grades analysed using atomic adsorption.

**Figure 5.8** QQ plots of the selected check samples against the in-situ NLZP samples
The check assays were submitted with QC samples to assess the quality of the analytical process. Subsequent to the analysis on blanks and CRM’s for Pb, Zn and Ag, the results of the QC samples showed acceptable repeatability with the historical results. Therefore, it is Snowden’s opinion that these results give confidence that the historical assays were accurate.
Figure 5.10  Analyses of blanks submitted with Northern Extension check assays

Blanks analyses for Pb

- Blanks
- Upper Limit

Blanks analyses for Zn

- Blanks
- Upper Limit
Figure 5.11 Analyses of AMIS0147 submitted with Northern Extension check assays
Figure 5.12  Analyses of AMIS0082 submitted with Northern Extension check assays

AMIS0082 analyses for Pb

AMIS0082 analyses for Zn

AMIS0082 analyses for Ag
Figure 5.13  Analyses of AMIS0158 submitted with Northern Extension check assays
Figure 5.14  Pb Analysis HARD Plot for the Northern Extension final check assays

Figure 5.15  Zn Analysis HARD Plot for the Northern Extension final check assays
Figure 5.16  Ag Analysis HARD Plot for the Northern Extension final check assays
Figure 5.17  Analyses of blanks submitted with South Mine Channel check assays

Blanks analyses for Pb

Blanks analyses for Zn
Figure 5.18 Analyses of AMIS0147 submitted with South Mine Channel check assays
Figure 5.19: Analyses of AMIS0082 submitted with South Mine Channel check assays

AMIS0082 analyses for Pb

AMIS0082 analyses for Zn

AMIS0082 analyses for Ag
Figure 5.20 Analyses of AMIS0158 submitted with South Mine Channel check assays

AMIS0158 analyses for Pb

AMIS0158 analyses for Zn

AMIS0158 analyses for Ag
Figure 5.21  Pb Analysis HARD Plot for the South Mine Channel final check assays

![Pb Analysis HARD Plot](image1)

Figure 5.22  Zn Analysis HARD Plot for the South Mine Channel final check assays

![Zn Analysis HARD Plot](image2)
Figure 5.23  Ag Analysis HARD Plot for the South Mine Channel final check assays
Figure 5.24 Analyses of blanks submitted with South Mine Junction check assays

Blanks analyses for Pb

<table>
<thead>
<tr>
<th>Pb %</th>
<th>A8719</th>
<th>A8752</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Blanks

Upper Limit

Blanks analyses for Zn

<table>
<thead>
<tr>
<th>Zn %</th>
<th>A8719</th>
<th>A8752</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Blanks

Upper Limit
Figure 5.25 Analyses of AMIS0147 submitted with South Mine Junction check assays
Figure 5.26 Analyses of AMIS0082 submitted with South Mine Junction check assays

AMIS0082 analyses for Pb

AMIS0082 analyses for Zn

AMIS0082 analyses for Ag
Figure 5.27  Pb Analysis HARD Plot for the South Mine Junction final check assays

Figure 5.28  Zn Analysis HARD Plot for the South Mine Junction final check assays
5.3 Bulk Density

A global density of 3.45 t/m³ was provided by NRR to be used with this Resource estimate for the ore. The density derived from the measurement of the density of 311 drilling and channel samples, selected within the ore envelopes, corresponds closely to this (Table 5.2).

Table 5.2 Density Measurements in drillhole and channel samples within the Resource Wireframes

<table>
<thead>
<tr>
<th>Drill Program</th>
<th>Description</th>
<th>Number of Samples</th>
<th>Type</th>
<th>Avg. Density (t/m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nam_Pb</td>
<td>Kalahari Underground Drilling (2008)</td>
<td>85</td>
<td>DD</td>
<td>3.6</td>
</tr>
<tr>
<td></td>
<td>Kalahari Underground Drilling (2008)</td>
<td>164</td>
<td>RC</td>
<td>3.6</td>
</tr>
<tr>
<td>UG_CH</td>
<td>NRR Underground Channel Sampling (2011 to 2013)</td>
<td>62</td>
<td>CH</td>
<td>3.2</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>311</strong></td>
<td></td>
<td><strong>3.5</strong></td>
</tr>
</tbody>
</table>
According to the NRR procedure (Appendix E), density measurements for exploration drilling are carried out at the mine on half diamond core samples using the immersion method, where a sample is weighed in dry air and when it is immersed in water. Density is then calculated using the following equation:

\[
\text{Dry bulk density} = \frac{W_1}{W_1 - (W_4 - (W_3))}
\]

Where,
- \(W_1\) is the sample dry weight
- \(W_2\) is the weight of each soaked sample
- \(W_3\) is the weight of contained water \((W_2 - W_1)\)
- \(W_4\) is the weight of sample and the harness in the water

To determine \(W_2\), the sample is left in water overnight to absorb the water before a weight under water can be determined. To assess precision of the measurements the first ten samples are repeated. According to the procedure, the density measurements should be done on all rock types, including waste. In Snowden’s opinion the process is adequate. There is no procedure for the bulk density measurements undertaken by Kalahari.

### 5.4 Database Integrity

The following sheets were exported from the database to prepare a desurveyed drillhole file (file containing a set of XYZ sample centre points, with lengths and directions which represent the hole traces) in CAE Datamine Studio 3™:

- DHCollars
- DHSurveys
- DHLithology
- DHSample
- DHSample_QAQC

The desurveyed drillhole file was checked for the following errors:

- Duplicates;
- Erroneous sample positions;
- Overlaps; and
- Missing information.
Only minor validation issues were identified, which comprised of 15 collar positions for planned holes which had not yet been drilled and some QA/QC information. Assays returning a value less than the detection limits have been given negative detection values in the database, e.g. a detection limit of 10 ppm was recorded as -10 ppm. These were reset to half the detection limit for modelling purposes. Further, not all the QA/QC information was saved in the DHSample_QAQC sheet; some QA/QC information was saved in the DHSample sheet.

5.5 Topography Validation

The surface profile at the project area was surveyed by African Geomatics. The topography of the NLZP was confirmed to be similar according to observations made during the site visit. The modelled surface corresponds to the drillhole information, namely the collar positions and the logged information (Figure 5.30 and Figure 5.31).

Figure 5.30  A north-easterly diagonal section showing the topography and collar positions
Figure 5.31  A 3D view showing the topography and collar positions
6 Geological Modelling and Domaining

The in-situ NLZP Resource model was built in the UTM coordinate system and a single kriging domain was selected, with dynamic anisotropy used for the estimation.

The three dimensional wireframe models of the mineralisation are based on sectional interpretations using CAE Datamine Studio 3™, based on assays (1.00% Pb + Zn cut-off), structural, geological and DHTEM data.

6.1 Lithology

The logging codes in the underground drillholes were used to model the major geological units. These include the ferruginous or sulphidic (FeS) lodes, pegmatite and dolerite. The major unit surrounding these is marble, which was not modelled.

![NLZP Lithology Model](image)

The geometry of the FeS lodes was used as an additional interpretation tool for the mineralised distribution.

6.2 Mineralised Envelopes

Mineralised envelopes were created to define the ore lodes (Pb % + Zn % > 1) that constitute the in-situ NLZP mineralised zones.
The exploration drillholes and channel samples were used to define the mineralisation. The shape and orientation of the ore lodes appeared an accurate reflection of the mineralisation trends shown in the drilling and sampling data.

The mineralised zone values are stored in a field called EZONE in the model. Blocks that fall within the mineralised envelopes have an EZONE value of 1 (ore). Blocks outside the mineralised envelopes have an EZONE value of 2 (waste).
7 Exploratory Data Analysis

Before undertaking the estimate, the data were analysed first, in order to understand how the estimate should be approached. Exploration and channel samples were used for the Resource estimate. High value samples that were excluded from the mineralised zones, but that were contiguous to the defined mineralised envelopes, were re-assigned to the appropriate mineralised kriging zones.

7.1 Bias testing

Bias testing for PbZn% (as used for defining the ore envelopes) was investigated for the selected ore portions of the Northern Extension (DD vs. RC) and South Mine (DD vs. Channel Samples) datasets. The statistical results are shown in Table 7.1 and Table 7.2.

<table>
<thead>
<tr>
<th>Northern Extension</th>
<th>DD (PbZn %)</th>
<th>RC (PbZn %)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Samples</td>
<td>85</td>
<td>368</td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Maximum</td>
<td>44.0</td>
<td>54.6</td>
</tr>
<tr>
<td>Mean</td>
<td>9.3</td>
<td>8.5</td>
</tr>
<tr>
<td>Std. Deviation</td>
<td>9.9</td>
<td>9.7</td>
</tr>
<tr>
<td>CoV</td>
<td>1.1</td>
<td>1.1</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>South Mine</th>
<th>DD (PbZn %)</th>
<th>Channel Samples (PbZn %)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Samples</td>
<td>134</td>
<td>466</td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Maximum</td>
<td>53.1</td>
<td>43.4</td>
</tr>
<tr>
<td>Mean</td>
<td>6.2</td>
<td>6.8</td>
</tr>
<tr>
<td>Std. Deviation</td>
<td>9.1</td>
<td>8.0</td>
</tr>
<tr>
<td>CoV</td>
<td>1.5</td>
<td>1.2</td>
</tr>
</tbody>
</table>

The percentage difference between the DD and RC for the Northern Extension PbZn % mean is 9%, the same as the difference between the DD and channel samples for the South Mine.

7.2 Compositing

All samples (exploration and channel) were composited to a 1 metre composite length.
7.3 Grade Capping

Grade capping was applied in order to lessen the effect of individual high grade samples in the estimate. In cases where individual samples would unduly influence the values of surrounding model cells, without the support of other high grade samples, a top cap was applied. The top limits that were applied to the EZONE=1 samples (ore) per element type, are listed in Table 7.3.

<table>
<thead>
<tr>
<th>Element</th>
<th>Grade Capping limits</th>
<th>Percentage Samples Capped</th>
<th>Mean (before capping)</th>
<th>Mean (after capping)</th>
<th>CV (before capping)</th>
<th>CV (after capping)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb %</td>
<td>23.05</td>
<td>0.22</td>
<td>2.07</td>
<td>2.02</td>
<td>1.9</td>
<td>1.8</td>
</tr>
<tr>
<td>Zn %</td>
<td>33.48</td>
<td>0.54</td>
<td>5.46</td>
<td>5.41</td>
<td>1.3</td>
<td>1.2</td>
</tr>
<tr>
<td>Ag g/t</td>
<td>331</td>
<td>0.55</td>
<td>42.18</td>
<td>41.21</td>
<td>1.5</td>
<td>1.4</td>
</tr>
</tbody>
</table>
7.4 Boundary Analysis

Mineralisation wireframes for the in-situ NLZP deposit were defined on a 1.00% (Pb % + Zn %) grade cut-off. The boundary between the ore and waste was analysed, in order to determine the accuracy of the modelling, as well as whether a hard or soft kriging approach was required. Hard boundaries are those which, during geostatistical analysis, indicate an abrupt change in average grade or variability at the contact between two domains, in this case, ore and waste (Figure 7.2). Based on the results of the boundary analyses, the mineralised boundary was deemed to be hard.

Figure 7.2 Mineralised boundary test graph for the in-situ NLZP deposit

7.5 Composite Statistics

The summary statistics for the Pb %, Zn % and Ag g/t values in the samples for the mineralised material (EZONE = 1) are shown in Table 7.4, followed by the associated histograms (Figure 7.3 to Figure 7.5). A top grade cap was applied as described in Section 7.3 above, to set the extreme grades to grades closer to the average grade of the population and thereby reducing the variability and coefficient of variance (CV) within the ore material.
Table 7.4  Summary statistics of the composited and capped Pb %, Zn % and Ag g/t ore sample values for the in-situ NLZP deposit

<table>
<thead>
<tr>
<th></th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Samples</td>
<td>923</td>
<td>930</td>
<td>923</td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0.01</td>
<td>0.25</td>
</tr>
<tr>
<td>Maximum</td>
<td>23.05</td>
<td>33.48</td>
<td>331</td>
</tr>
<tr>
<td>Mean</td>
<td>2.02</td>
<td>5.41</td>
<td>41.21</td>
</tr>
<tr>
<td>Variance</td>
<td>13.9</td>
<td>43.5</td>
<td>3237</td>
</tr>
<tr>
<td>Std. Deviation</td>
<td>3.7</td>
<td>6.6</td>
<td>56.9</td>
</tr>
<tr>
<td>CV</td>
<td>1.8</td>
<td>1.2</td>
<td>1.4</td>
</tr>
</tbody>
</table>

Figure 7.3  Histogram for the composited Pb % for the defined mineralised in-situ NLZP deposit
Figure 7.4  Histogram for the composited Zn % for the defined mineralised in-situ NLZP deposit

Histogram for capped Zn % (EZONE 1)

Total Samples : 930
Minimum      : 0.012
Maximum      : 33.48
Mean         : 5.41
Variance     : 43.54
StdDeviation : 6.60
Coeff.Variation : 1.22
7.6 Stationarity

The assumption of stationarity is a statistical concept where data that has been pooled within a given area or domain is geologically homogenous with the same statistical properties, i.e. the mean and variance of values do not depend on location.

Stationarity tests for Pb %, Zn % and Ag g/t, for the in-situ NLZP deposit composited and capped drillhole data, from south to north and west to east, were undertaken for the ore (EZONE = 1). The data was separated into the Northern Extension and the South Mine. Some instances of local variability in the grades were identified, which is most likely a result of the data density.
Figure 7.6  Stationarity test results for the composited Pb % for the defined mineralised in-situ NLZP deposit – Northern Extension

Northern Extension – Pb % on Northing

Northern Extension – Pb % on Easting
Figure 7.7  Stationarity test results for the composited Zn % for the defined mineralised in-situ NLZP deposit – Northern Extension
Figure 7.8 Stationarity test results for the composited Ag g/t for the defined mineralised in-situ NLZP deposit – Northern Extension
Figure 7.9  Stationarity test results for the composited Pb % for the defined mineralised in-situ NLZP deposit – South Mine
Figure 7.10  Stationarity test results for the composited Zn % for the defined mineralised in-situ NLZP deposit – South Mine

South Mine – Zn % on Northing

South Mine – Zn % on Easting
Figure 7.11 Stationarity test results for the composited Ag g/t for the defined mineralised in-situ NLZP deposit – South Mine
Variography

Composite data from the Northern Extension underground drillhole dataset was used to generate experimental variograms for Pb, Zn and Ag, using Supervisor 8.

Experimental variograms were investigated at ten degree increments, with a lag ranging between 10 and 20 along the strike and dip directions. The nugget effect was modelled using the downhole variogram, where the lag was set equal to the composite length of 1 m. The variogram parameters used for the estimation in CAE Datamine Studio 3<sup>TM</sup> for the different elements are listed in Table 8.1.

Table 8.1 Variogram parameters used for the ore grade estimations

<table>
<thead>
<tr>
<th>Element</th>
<th>Angle1</th>
<th>Angle2</th>
<th>Angle3</th>
<th>Axis1</th>
<th>Axis2</th>
<th>Axis3</th>
<th>Nugget (C0)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>35</td>
<td>125</td>
<td>0</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>0.09</td>
</tr>
<tr>
<td>Zn</td>
<td>35</td>
<td>125</td>
<td>110</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>0.17</td>
</tr>
<tr>
<td>Ag</td>
<td>35</td>
<td>125</td>
<td>-45</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>0.08</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Element</th>
<th>Structure 1</th>
<th>Structure 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Range X (m)</td>
<td>Range Y (m)</td>
</tr>
<tr>
<td>Pb</td>
<td>36</td>
<td>46</td>
</tr>
<tr>
<td>Zn</td>
<td>39</td>
<td>60</td>
</tr>
<tr>
<td>Ag</td>
<td>40</td>
<td>44</td>
</tr>
</tbody>
</table>

Figure 8.1 Experimental variograms used for the Pb % in-situ NLZP deposit
Figure 8.2  Experimental variograms used for the Zn % in-situ NLZP deposit

Figure 8.3  Experimental variograms used for the Ag g/t in-situ NLZP deposit
9 Kriging Neighbourhood Analysis (KNA)

The search ranges were optimised in order to minimise the kriging variance. Negative kriging weights are used, whereas octants search was not used. The optimum grid system was selected as the point at which no significant improvement was made in reducing the average covariance (measure of how much two variables change together) between pairs of points. The optimum grid was: 3 points in the X direction, 3 points in the Y direction and 3 points in the Z direction.

9.1 Block Size and Discretisation

The results of the KNA study recommended a parent block size of 4 mE by 4 mN by 4 mRL for all elements, as well as discretisation of 3 by 3 by 3.
Figure 9.1  Optimal block sizes for Pb, Zn and Ag in-situ NLZP deposit

- Block size optimisation Pb
- Block size optimisation Zn
- Block size optimisation Ag
Figure 9.2 Discretisation parameters Pb, Zn and Ag in-situ NLZP deposit
### 9.2 Search Range

The optimal search ranges to be used for the elements are shown in Table 9.1 and Figure 9.3.

These search distances take into account the average ranges for the three estimated elements as determined with variography. The experimental semi-variograms data were used in conjunction with the discretisation parameters and block sizes to determine the appropriate search volumes.

<table>
<thead>
<tr>
<th>Element</th>
<th>Search X (m)</th>
<th>Search Y (m)</th>
<th>Search Z (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>25</td>
<td>45</td>
<td>5</td>
</tr>
<tr>
<td>Zn</td>
<td>25</td>
<td>45</td>
<td>5</td>
</tr>
<tr>
<td>Ag</td>
<td>25</td>
<td>45</td>
<td>5</td>
</tr>
</tbody>
</table>

Table 9.1 A summary of the optimal search ranges for Pb %, Zn % and Ag g/t for the in-situ NLZP deposit
Figure 9.3  Neighbourhood search optimisation for Pb, Zn and Ag in-situ NLZP deposit
9.3 Number of Composites

A minimum of 4 and a maximum of 20 composite samples for Pb, Zn and Ag were used for the Resource ore estimation.

Figure 9.4 Number of composites for Pb, Zn and Ag in-situ NLZP deposit
10 Grade Estimation

The model was estimated using Ordinary Kriging into the ore zone (EZONE = 1). Block sizes were 4 m x 4 m x 4 m (X Y Z) and, where appropriate, selective sub-celling was used for definition on the mineralisation boundaries. The underground exploration and channel samples were used for the estimation and the kriged Pb, Zn and Ag values for each block were stored in the PB, ZN and AG fields.

10.1 Block Modelling

For the block model, a parent cell size of 4 mE by 4 mN by 4 mRL, with 4 sub-cells in the X, Y and Z directions to fill the narrower mineralisation wireframes, were used. The block model was not rotated and covers the entire extent of the in-situ NLZP deposit. The parameters used for the model prototype are summarised in the following table.

<table>
<thead>
<tr>
<th>Field</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>XMORIG</td>
<td>474600</td>
</tr>
<tr>
<td>YMORIG</td>
<td>7509500</td>
</tr>
<tr>
<td>ZMORIG</td>
<td>-90</td>
</tr>
<tr>
<td>XINC</td>
<td>4</td>
</tr>
<tr>
<td>YINC</td>
<td>4</td>
</tr>
<tr>
<td>ZINC</td>
<td>4</td>
</tr>
<tr>
<td>NX</td>
<td>230</td>
</tr>
<tr>
<td>NY</td>
<td>190</td>
</tr>
<tr>
<td>NZ</td>
<td>110</td>
</tr>
</tbody>
</table>

10.2 Search Strategy

The search parameters that were used for the estimation for the ore kriging zone were deduced from the results of the KNA study. A three phased search pass was applied and the orientation of the search ellipsoid was aligned to the modelled variography. This process involves the estimation being performed three times, where two expansion factors are used. These factors increase the size of the search ellipse used to select samples during each estimation run. This method ensures that blocks which are not estimated and populated with a grade value in the first run, are populated in one of the subsequent runs. The search volume applicable to the estimated grade within each block is recorded in the model and considered when classifying the Resource.
The ore (EZONE = 1) was estimated using dynamic anisotropy. This process allows the rotation angles for the search ellipsoid to be defined individually for each cell in the model, so that the search ellipsoid is aligned with the axes of mineralisation. This therefore requires the rotation angles to be interpolated into the model cells, which in turn requires a set of angles as the input data file for interpolation. The dip and dip direction of the major axis of anisotropy were defined by digitising strings in section perpendicular to the strike of the orebody. These strings were converted to points that contained the true dip and dip direction of the mineralisation (fields SANGLE1_F and SANGLE2_F in the search parameter file – see Table 10.2 to Table 10.4).

The search distances and angles take into account the average ranges and angles for the four estimated elements as determined with variography. The experimental semi-variograms data were used in conjunction with the discretisation parameters and block sizes to determine the appropriate search volumes.

### Table 10.2  Search criteria used for primary dynamic anisotropic estimation

<table>
<thead>
<tr>
<th>Element</th>
<th>Search Method</th>
<th>Search X</th>
<th>Search Y</th>
<th>Search Z</th>
<th>Search angle 1</th>
<th>Search angle 2</th>
<th>Search angle 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>1</td>
<td>25</td>
<td>45</td>
<td>5</td>
<td>35</td>
<td>125</td>
<td>0</td>
</tr>
<tr>
<td>Zn</td>
<td>1</td>
<td>25</td>
<td>45</td>
<td>5</td>
<td>35</td>
<td>125</td>
<td>110</td>
</tr>
<tr>
<td>Ag</td>
<td>1</td>
<td>25</td>
<td>45</td>
<td>5</td>
<td>35</td>
<td>125</td>
<td>45</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Element</th>
<th>Search axis 1</th>
<th>Search axis 2</th>
<th>Search axis 3</th>
<th>Min. Sample</th>
<th>Max. Sample</th>
<th>SANGL1_F</th>
<th>SANGL2_F</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>4</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Zn</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>4</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Ag</td>
<td>Z</td>
<td>X</td>
<td>Z</td>
<td>4</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
</tbody>
</table>

The true dip and dip direction were interpolated into the block model. For the Pb, Zn and Ag estimations, the search volumes were oriented individually for each cell, using the values in the TRDIPDIR and TRDIP fields. The lengths of the axes of the search volume are fixed for each search volume, as defined by fields Search X, Search Y and Search Z.

In the instances were not estimated after the primary dynamic anisotropy run, a second estimation run was initiated, and thereafter a third, with the parameters as shown in Table 10.3 and Table 10.4.

### Table 10.3  Search criteria used for secondary dynamic anisotropic estimation

<table>
<thead>
<tr>
<th>Element</th>
<th>Search Method</th>
<th>Search X</th>
<th>Search Y</th>
<th>Search Z</th>
<th>Search angle 1</th>
<th>Search angle 2</th>
<th>Search angle 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>1</td>
<td>37.5</td>
<td>67.5</td>
<td>7.5</td>
<td>35</td>
<td>125</td>
<td>0</td>
</tr>
<tr>
<td>Zn</td>
<td>1</td>
<td>37.5</td>
<td>67.5</td>
<td>7.5</td>
<td>35</td>
<td>125</td>
<td>110</td>
</tr>
<tr>
<td>Ag</td>
<td>1</td>
<td>37.5</td>
<td>67.5</td>
<td>7.5</td>
<td>35</td>
<td>125</td>
<td>45</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Element</th>
<th>Search axis 1</th>
<th>Search axis 2</th>
<th>Search axis 3</th>
<th>Min. Sample</th>
<th>Max. Sample</th>
<th>SANGL1_F</th>
<th>SANGL2_F</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>3</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Zn</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>3</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Ag</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>3</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Element</td>
<td>Search Method</td>
<td>Search X</td>
<td>Search Y</td>
<td>Search Z</td>
<td>Search angle 1</td>
<td>Search angle 2</td>
<td>Search angle 3</td>
</tr>
<tr>
<td>---------</td>
<td>---------------</td>
<td>----------</td>
<td>----------</td>
<td>----------</td>
<td>----------------</td>
<td>----------------</td>
<td>----------------</td>
</tr>
<tr>
<td>Pb</td>
<td>1</td>
<td>50</td>
<td>90</td>
<td>10</td>
<td>35</td>
<td>125</td>
<td>0</td>
</tr>
<tr>
<td>Zn</td>
<td>1</td>
<td>50</td>
<td>90</td>
<td>10</td>
<td>35</td>
<td>125</td>
<td>110</td>
</tr>
<tr>
<td>Ag</td>
<td>1</td>
<td>50</td>
<td>90</td>
<td>10</td>
<td>35</td>
<td>125</td>
<td>-45</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Element</th>
<th>Search axis 1</th>
<th>Search axis 2</th>
<th>Search axis 3</th>
<th>Min. Sample</th>
<th>Max. Sample</th>
<th>SANGL1_F</th>
<th>SANGL2_F</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>1</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Zn</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>1</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
<tr>
<td>Ag</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>1</td>
<td>20</td>
<td>TRDIPDIR</td>
<td>TRDIP</td>
</tr>
</tbody>
</table>

To distinguish between the ore estimation runs for classification purposes, the SVOL_PB, SVOL_ZN and SVOL_AG were coded as follows:

<table>
<thead>
<tr>
<th>SVOL_PB</th>
<th>SVOL_ZN</th>
<th>SVOL_AG</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>-</td>
<td>-</td>
<td>-</td>
<td>Absent – Not Estimated</td>
</tr>
<tr>
<td>1</td>
<td>1</td>
<td>1</td>
<td>Primary Estimation Run</td>
</tr>
<tr>
<td>2</td>
<td>2</td>
<td>2</td>
<td>Secondary Estimation Run</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>3</td>
<td>Tertiary Estimation Run</td>
</tr>
</tbody>
</table>

Any ore blocks that were still not estimated after the third search, were given a default value of 0.001 for Pb, Zn and Ag. The waste (EZONE = 2) was not estimated, but all waste blocks were given a default value of 0.001 for Pb, Zn and Ag for mine planning purposes.

10.3 Block Model Fields

Fields used in the block model are summarised and explained in the following tables.
### Table 10.6  Block model fields

<table>
<thead>
<tr>
<th>Field/s</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>IJK</td>
<td>Block index</td>
</tr>
<tr>
<td>XC, YC, ZC</td>
<td>Sub-cell coordinates</td>
</tr>
<tr>
<td>XINC, YINC, ZINC</td>
<td>Size of sub-cells</td>
</tr>
<tr>
<td>EZONE</td>
<td>Kriging zone used in estimation</td>
</tr>
<tr>
<td>DENSITY</td>
<td>Density</td>
</tr>
<tr>
<td>AREA</td>
<td>Area subdivision</td>
</tr>
<tr>
<td>TRDIPDIR</td>
<td>True Dip Direction from section strings (used for ore estimation)</td>
</tr>
<tr>
<td>TRDIP</td>
<td>True Dip from section strings (used for ore estimation)</td>
</tr>
<tr>
<td>PB</td>
<td>Estimated lead (Pb) [%]</td>
</tr>
<tr>
<td>ZN</td>
<td>Estimated zinc (Zn) [%]</td>
</tr>
<tr>
<td>AG</td>
<td>Estimated silver (Ag) [g/t]</td>
</tr>
<tr>
<td>SVOL_PB</td>
<td>Pb Estimation Run Indicator</td>
</tr>
<tr>
<td>SVOL_ZN</td>
<td>Zn Estimation Run Indicator</td>
</tr>
<tr>
<td>SVOL_AG</td>
<td>Ag Estimation Run Indicator</td>
</tr>
<tr>
<td>MINED</td>
<td>Mined out indicator</td>
</tr>
<tr>
<td>PbZn</td>
<td>Sum of Pb and Zn per block [%]</td>
</tr>
<tr>
<td>XMORIG, YMORIG, ZMORIG</td>
<td>Block model origin (bottom left corner)</td>
</tr>
<tr>
<td>NX, NY, NZ</td>
<td>Number of parent cells in X, Y and Z (4 m x 4 m x 4 m)</td>
</tr>
</tbody>
</table>

### Table 10.7  Summary of definition of the EZONE codes used in the Resource model

<table>
<thead>
<tr>
<th>EZone</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Ore</td>
</tr>
<tr>
<td>2</td>
<td>Waste</td>
</tr>
</tbody>
</table>

### Table 10.8  Summary of definition of the AREA codes used in the Resource model

<table>
<thead>
<tr>
<th>Area</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>North</td>
<td>Northern Extension</td>
</tr>
<tr>
<td>South</td>
<td>South Mine</td>
</tr>
</tbody>
</table>

### Table 10.9  Summary of definition of the MINED codes used in the Resource model

<table>
<thead>
<tr>
<th>Mined</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>Not Mined</td>
</tr>
<tr>
<td>1</td>
<td>Mined</td>
</tr>
</tbody>
</table>
11 Model Validation

In order to validate the estimated kriged block results in the Resource model, the model was compared against the composite drillhole and channel sample data. A number of techniques were used for the validation. These included visual validation of block grades to input composite drillhole and channel sample files, global comparisons between average block model grade and average composite sample grade, and slicing plots through the deposit in northing and easting, comparing average block model grades with average sample grades for each slice.

11.1 Visual Validation

Section slices were inspected to see if the sample grades and model grades are comparable, to assess whether the estimation used local data in order to assign a grade. Figure 11.1 to Figure 11.6 show examples of sectional slices with the block model and composite samples coloured on Pb, Zn and Ag grades, respectively. They show that the areas of high grade in the samples seem to align with the grades in the block model.

Figure 11.1  Visual validation of Pb % in the Northern Extension of NLZP
Figure 11.2 Visual validation of Zn % in the Northern Extension of NLZP

Figure 11.3 Visual validation of Ag g/t in the Northern Extension of NLZP
Figure 11.4  Visual validation of Pb % in the South Mine of NLZP

Figure 11.5  Visual validation of Zn % in the South Mine of NLZP
11.2 Global Mean Comparison

The global mean grade comparison between each element in the ore model and the composite sample file from which the blocks were estimated are shown in Table 11.1. In general, the model validates well, with similar mean grades in comparison to the samples.

Table 11.1 Global Mean Grade Comparison for the in-situ NLZP

<table>
<thead>
<tr>
<th>Element</th>
<th>Model Mean</th>
<th>Comp. Mean</th>
<th>Diff. to Comp. Mean</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb %</td>
<td>2.14</td>
<td>2.02</td>
<td>6%</td>
</tr>
<tr>
<td>Zn %</td>
<td>5.21</td>
<td>5.41</td>
<td>-4%</td>
</tr>
<tr>
<td>Ag g/t</td>
<td>42.57</td>
<td>41.21</td>
<td>3%</td>
</tr>
</tbody>
</table>

11.3 Validation Slices

As part of the validation process, the block model and input samples that fall within defined sectional criteria were compared and the results displayed graphically to check for visual discrepancies between grades.

The validation plots comparing sample grades to model grades for the ore in the Northern Extension and the South Mine are presented in Figure 11.7 to Figure 11.12. They show good correlation for all the elements between samples and model. Generally the model does as it should and follows the general pattern of the sample grade, with some smoothing of higher and lower grades.
Figure 11.7 Validation plots for the composited Pb % drillhole data vs. kriged model data for the defined mineralised in-situ NLZP deposit – Northern Extension
Figure 11.8  Validation plots for the composited Zn % drillhole data vs. kriged model data for the defined mineralised in-situ NLZP deposit – Northern Extension
Figure 11.9  Validation plots for the composited Ag g/t drillhole data vs. kriged model data for the defined mineralised in-situ NLZP deposit – Northern Extension
Figure 11.10 Validation plots for the composited Pb % channel sample data vs. kriged model data for the defined mineralised in-situ NLZP deposit – South Mine
Figure 11.11 Validation plots for the composited Zn % channel sample data vs. kriged model data for the defined mineralised in-situ NLZP deposit – South Mine
The summary statistics for the Pb, Zn and Ag values in the estimated blocks for the mineralised ore are shown in the table below, with the associated histograms.

Figure 11.12 Validation plots for the composites Ag g/t channel sample data vs. kriged model data for the defined mineralised in-situ NLZP deposit – South Mine
Table 11.2  Summary statistics of the Pb, Zn and Ag sample values in the estimated blocks

<table>
<thead>
<tr>
<th></th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Samples</td>
<td>148,173</td>
<td>148,173</td>
<td>148,173</td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
<td>0.1</td>
<td>0.2</td>
</tr>
<tr>
<td>Maximum</td>
<td>14.9</td>
<td>29.6</td>
<td>193.8</td>
</tr>
<tr>
<td>Mean</td>
<td>2.14</td>
<td>5.21</td>
<td>42.57</td>
</tr>
<tr>
<td>Variance</td>
<td>5.3</td>
<td>16.6</td>
<td>886.6</td>
</tr>
<tr>
<td>Std. Deviation</td>
<td>2.3</td>
<td>4.1</td>
<td>29.8</td>
</tr>
<tr>
<td>COV</td>
<td>1.1</td>
<td>0.8</td>
<td>0.7</td>
</tr>
</tbody>
</table>

Figure 11.13  Histogram for the estimated Pb % blocks for the defined mineralised in-situ NLZP deposit

Histogram for Pb % (EZONE 1)

- Total Samples: 148173
- Minimum: 0.001
- Maximum: 14.94
- Mean: 2.14
- Variance: 5.26
- StdDeviation: 2.29
- Coeff. Variation: 1.07
Figure 11.14 Histogram for the estimated Zn % blocks for the defined mineralised in-situ NLZP deposit

Histogram for Zn % (EZONE 1)

- Total Samples: 148173
- Minimum: 0.138
- Maximum: 29.62
- Mean: 5.21
- Variance: 16.62
- StdDeviation: 4.08
- Coeff. Variation: 0.78

Frequency vs. Zn %
Figure 11.15  Histogram for the estimated Ag g/t blocks for the defined mineralised in-situ NLZP deposit

Histogram for Ag g/t (EZONE 1)

<table>
<thead>
<tr>
<th>Metric</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Samples</td>
<td>148173</td>
</tr>
<tr>
<td>Minimum</td>
<td>0.231</td>
</tr>
<tr>
<td>Maximum</td>
<td>193.77</td>
</tr>
<tr>
<td>Mean</td>
<td>42.57</td>
</tr>
<tr>
<td>Variance</td>
<td>886.55</td>
</tr>
<tr>
<td>StdDeviation</td>
<td>29.77</td>
</tr>
<tr>
<td>Coeff. Variation</td>
<td>0.70</td>
</tr>
</tbody>
</table>
12 Mineral Resource Classification and Tabulation

This Mineral Resource has been classified in accordance with the JORC (2012) guidelines. The code sets out minimum standards, recommendations and guidelines for Public Reporting.

When following the guidelines of the JORC (2012) code, tonnage and grade estimates are classified so as to reflect different levels of geological confidence and different degrees of technical and economic evaluation. A geologist will estimate the Mineral Resource using geoscientific information such as drillhole cores, sample assay values and QA/QC data (Figure 12.1).

Figure 12.1 General relationship between Exploration Results, Mineral Resources and Ore Reserves

12.1 Classification

The estimated Mineral Resource has been classified according to the knowledge and confidence of the geological information.

Classification wireframes were created in plan view sections of 10 m intervals using CAE Datamine Studio 3™, and verified in vertical view sections of 15 m intervals. Areas were wireframed as Indicated where drillhole and channel sample spacing was generally within 15 m (X) by 40 m (Y) by 10 m (Z). Inferred wireframes were created for areas where the drillhole and channel sample grid spacing was at least 30 m (X) by 50 m (Y) by 15 m (Z). Additionally, for blocks to be classified as either Indicated or Inferred, a general geological continuity should be shown. This was determined by the variography and the search volumes calculated from the variogram ranges. As discussed in Section 10.2, three search volumes were used, orientated along the strike, dip direction and the angle of dip of the orebody. Blocks that were estimated within the primary search volume, generally show geological continuity. Blocks estimated within the secondary search volume, cannot be classified as Indicated, but at the most as Inferred. Blocks estimated within the tertiary search volume can only be classified as Blue Sky, a classification used for internal purposes.
Only ore blocks were classified, since the waste was not estimated. The following classification codes were applied in the Resource block model:

**Table 12.1 Description of Classification codes used in the Resource model**

<table>
<thead>
<tr>
<th>Class</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>Not Estimated</td>
</tr>
<tr>
<td>1</td>
<td>Measured – not present in model</td>
</tr>
<tr>
<td>2</td>
<td>Indicated</td>
</tr>
<tr>
<td>3</td>
<td>Inferred</td>
</tr>
<tr>
<td>4</td>
<td>Blue Sky</td>
</tr>
</tbody>
</table>

Two temporary fields were created in the Resource model for the purpose of classification coding. These were XCLASS and SVOL, where XCLASS refers to blocks falling within either the Indicated or Inferred wireframes, and SVOL, referring to blocks estimated within the primary, secondary or tertiary search volumes. The classification coding logic applied to the ore blocks were as follows:

- Blocks selected within the Indicated wireframe: XCLASS = 2
- Blocks selected within the Inferred wireframe: XCLASS = 3
- Blocks outside the Inferred wireframe: XCLASS = 0
- Blocks estimated within the primary search volume: SVOL = 1
- Blocks estimated within the secondary search volume: SVOL = 2
- Blocks estimated within the tertiary search volume: SVOL = 3

Blocks were assigned a CLASS value according to the coding logic as shown in Table 12.2.

**Table 12.2 Classification coding logic used in the Resource model**

<table>
<thead>
<tr>
<th>SVOL</th>
<th>XCLASS</th>
<th>CLASS</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2</td>
<td>2 (Indicated)</td>
</tr>
<tr>
<td>1</td>
<td>3</td>
<td>3 (Inferred)</td>
</tr>
<tr>
<td>1</td>
<td>0</td>
<td>4 (Blue Sky)</td>
</tr>
<tr>
<td>2</td>
<td>2</td>
<td>3 (Inferred)</td>
</tr>
<tr>
<td>2</td>
<td>3</td>
<td>3 (Inferred)</td>
</tr>
<tr>
<td>2</td>
<td>0</td>
<td>4 (Blue Sky)</td>
</tr>
<tr>
<td>3</td>
<td>2</td>
<td>4 (Blue Sky)</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>4 (Blue Sky)</td>
</tr>
<tr>
<td>3</td>
<td>0</td>
<td>4 (Blue Sky)</td>
</tr>
</tbody>
</table>
No Measured Resources have been declared for the in-situ NLZP deposit, as the geological and structural continuity is not clearly understood yet, and more information from a tighter spacing of diamond drilling is needed.
12.2 Depletion and Mineral Resource Tabulation

The 2013 NLZP in-situ Resource model was depleted by the mined out stopes, the developments and the areas of backfill. The classified, depleted NLZP in-situ Mineral Resource is reported above a (Pb + Zn) > 1.00% cut-off grade in Table 12.3.

Table 12.3  In-situ NLZP Classified Mineral Resource Estimate - Depleted

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>*Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>Northern Extension</td>
<td>529,000</td>
<td>3.45</td>
<td>2.8</td>
<td>5.4</td>
<td>48.2</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>130,000</td>
<td>3.45</td>
<td>1.9</td>
<td>4.3</td>
<td>41.6</td>
</tr>
<tr>
<td>Inferred</td>
<td>Northern Extension</td>
<td>251,000</td>
<td>3.45</td>
<td>1.8</td>
<td>7.2</td>
<td>38.9</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>7,000</td>
<td>3.45</td>
<td>2.1</td>
<td>3.5</td>
<td>52.6</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>917,000</td>
<td>3.45</td>
<td>2.4</td>
<td>5.7</td>
<td>44.8</td>
</tr>
</tbody>
</table>

*Tonnages have been rounded to the nearest 1000 t to reflect an estimate

Potential future opportunities were highlighted by the category Blue Sky, which are portions of the Mineral Resource that were found to fall outside the classification criteria, but were modelled and incorporated into the Mineral Resource. This category is included for internal use only. The classified, depleted NLZP in-situ Blue Sky estimate is reported above a (Pb + Zn) > 1.00% cut-off grade in Table 12.4.

Table 12.4  In-situ NLZP Blue Sky Estimate - Depleted

<table>
<thead>
<tr>
<th>Class</th>
<th>Area</th>
<th>*Tonnes</th>
<th>Density (t/m³)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>**Blue Sky</td>
<td>Northern Extension</td>
<td>55,000</td>
<td>3.45</td>
<td>1.1</td>
<td>11.5</td>
<td>36.9</td>
</tr>
<tr>
<td></td>
<td>South Mine</td>
<td>3,000</td>
<td>3.45</td>
<td>0.9</td>
<td>5.5</td>
<td>29.1</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>58,000</td>
<td>3.45</td>
<td>1.1</td>
<td>11.2</td>
<td>36.5</td>
</tr>
</tbody>
</table>

*Tonnages have been rounded to the nearest 1000 t to reflect an estimate

**Blue Sky Classification is for Internal Purposes only and is not reportable under JORC (2012) guidelines as Resources
13 Model Reconciliation

The Northern Extension ore lodes were re-interpreted, incorporating structural and geological data, as well as DHTEM plates. Additional historical drillholes were used for the interpretation as indication to the continuity of the ore lodes. The South Mine area showed an increase in the modelled volume due to additional drillhole and channel sampling data.

Grade tonnage curves and values of PbZn (Pb % + Zn %) ore blocks for the NLZP in-situ deposit, regardless of classification, are shown in Figure 13.1, Figure 13.2 and Table 13.1 below. The 2012 CSA and the 2013 Snowden in-situ NLZP models were used for the calculation of comparative grade tonnage curves and percentages at various cut-offs.

Figure 13.1 PbZn Grade-tonnage curves (Tonnes above cut-off vs. Cut-off grade) for the in-situ NLZP deposit - Undepleted [Blue – 2012; Red – 2013]
Figure 13.2  PbZn Grade-tonnage curves (Grade above cut-off vs. Cut-off grade) for the in-situ NLZP deposit - Undepleted [Blue – 2012; Red – 2013]

Table 13.1  Comparative PbZn Mineral Resources for in-situ NLZP 2012 and 2013 at various cut-off grades - Undepleted

<table>
<thead>
<tr>
<th>Cut-off</th>
<th>2012</th>
<th>2013</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>*Tonnes</td>
<td>PbZn (%)</td>
</tr>
<tr>
<td>0</td>
<td>699,000</td>
<td>8.7</td>
</tr>
<tr>
<td>1.0</td>
<td>689,000</td>
<td>8.8</td>
</tr>
<tr>
<td>2.5</td>
<td>595,000</td>
<td>9.9</td>
</tr>
<tr>
<td>5.0</td>
<td>449,000</td>
<td>11.9</td>
</tr>
<tr>
<td>10.0</td>
<td>233,000</td>
<td>16.1</td>
</tr>
<tr>
<td>15.0</td>
<td>106,000</td>
<td>20.9</td>
</tr>
<tr>
<td>20.0</td>
<td>52,000</td>
<td>24.9</td>
</tr>
<tr>
<td>25.0</td>
<td>21,000</td>
<td>28.7</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1000 t to reflect an estimate
**These values are for the entire models, regardless of classification
The model reconciliation between the 2012 CSA and 2013 Snowden Resource models were done on the in-situ NLZP. Table 13.2 details the reconciliations of the undepleted, unclassified Resources and are quoted at a PbZn % cut-off grade of 1.00%.

Table 13.2  In-situ 2012 CSA and 2013 Snowden NLZP Mineral Resources comparison reported at a cut-off PbZn % > 1 (Undepleted)

<table>
<thead>
<tr>
<th>Model</th>
<th>Cut-off PbZn (%)</th>
<th>*Tonnes</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2012</td>
<td>1.00</td>
<td>689,000</td>
<td>2.38</td>
<td>6.41</td>
<td>44.61</td>
</tr>
<tr>
<td>2013</td>
<td>1.00</td>
<td>928,000</td>
<td>2.4</td>
<td>5.7</td>
<td>44.9</td>
</tr>
</tbody>
</table>

* Tonnages have been rounded to the nearest 1000 t to reflect an estimate
**Blue Sky classification numbers are not reported

The increase in tonnage is due to the increase in volume for the modelled ore envelopes from 2012 to 2013. This was a result of re-interpretation and using historic drillhole data giving confidence in continuity of mineralisation in the Northern Extension and the incorporation of new drilling and channel sampling results in the South Mine.

Table 13.3  Comparison between In-situ 2012 and 2013 NLZP Mineral Resource Wireframe Volumes - Undepleted

<table>
<thead>
<tr>
<th>Model</th>
<th>*Volume</th>
<th>Density (t/m³)</th>
<th>*Tonnes</th>
</tr>
</thead>
<tbody>
<tr>
<td>2012 Northern Extension</td>
<td>175,000</td>
<td>3.45</td>
<td>603,000</td>
</tr>
<tr>
<td>2013 Northern Extension</td>
<td>247,000</td>
<td>3.45</td>
<td>853,000</td>
</tr>
<tr>
<td>2012 South Mine</td>
<td>34,000</td>
<td>3.45</td>
<td>116,000</td>
</tr>
<tr>
<td>2013 South Mine</td>
<td>43,000</td>
<td>3.45</td>
<td>149,000</td>
</tr>
<tr>
<td>2012 Total</td>
<td>209,000</td>
<td>3.45</td>
<td>720,000</td>
</tr>
<tr>
<td>2013 Total</td>
<td>290,000</td>
<td>3.45</td>
<td>1,002,000</td>
</tr>
</tbody>
</table>

*Volumes and Tonnages have been rounded to the nearest 1000
14 Reasonable Prospects for Eventual Economic Extraction

14.1 Summary

Any Mineral Resource reported in accordance with The JORC Code (2012) is required to have reasonable prospects for eventual economic extraction. This reasonableness covers both mining and processing aspects, and anything else material that might be considered during the evaluation of a resource.

It is concluded that the Namib in-situ resource is a viable underground proposition and can be processed effectively (Stirling, 2013).

Mining will be via an underground mechanised operation utilising open stoping. It is expected that mining will be at a rate of 250,000 t per annum. The in-situ waste will be backfilled into existing underground voids. The tailings would be placed into the newer tailings dam.

The assumptions for the lead and zinc metal price are taken at a discount to today's prices.

14.2 Metallurgical Testwork – In-situ ore

Mineralogical studies note the presence of occluded iron in sphalerite, however the tests indicated that acceptable zinc concentrates could be achieved by milling to 80% passing 45 microns and flotation. Galena recovery and lead concentrate grades presented no problems. Sphalerite liberation at 45 microns was 89%.

Conclusions from the zinc concentrate upgrading tests are summarised as:

- The pyrite and pyrrhotite can be successfully depressed by using polyacrylamides as well as pH control resulting in the selective recovery of the sphalerite;
- Regrinding the zinc rougher concentrate to a P80 of 45 micron prior to cleaning produced zinc concentrate grades of in excess of 50% zinc at a recovery of about 85%.

Two beneficiation options are available to produce the zinc concentrate:

- The iron sulphide minerals can either be depressed in the rougher circuit, or
- A separate bulk zinc/iron sulphide could be generated, then selectively recover a zinc concentrate by depressing the iron sulphides. This option will produce a saleable pyrite-pyrrhotite concentrate.

An average silver recovery of about 70% was achieved in the lead rougher concentrate with a grade of about 600 g/t Ag. After cleaning the silver grades increased to over a 1,000 g/t Ag, at lower recoveries. However, the test was again performed in an open circuit, in a full scale plant the cleaner tails will be re-circulated.

Results demonstrated and confirmed that the NLZP ore can successfully be beneficiated to produce high grade lead and zinc concentrates with the silver associated with the lead.
14.3 Process Design and Flowsheet

A flowsheet is proposed that incorporates two-stage crushing followed by ball milling and sequential lead and zinc sulphide flotation. A conservative approach has been taken in the Scoping Study (Sterling, 2013) with respect to the ore metal recoveries and grades used to determine costs of production.

- Recoveries of Zn 85%, Pb 85% and Ag 70% are taken.
- Concentrate grades of 52% Zn and 55% Pb are taken.

The flowsheet for the in-situ ore will essentially follow that proposed in Stirling (2013) – Figure 14.1.

Figure 14.1 Recommended ore treatment flow sheet
15 References


Appendix A  JORC TABLE 1
### Section 1 of the JORC Table 1 for the in-situ NLZP

<table>
<thead>
<tr>
<th>Section 1: Sampling Techniques and Data Criteria</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Sampling techniques</strong></td>
<td>34 DD, 111 RC and 19 percussion (PERC) holes were drilled at an overall 15 m by 40 m spacing, and samples were collected at 1 m intervals over mineralised intersection, from which 2 - 2.5 kg of sample was collected for analysis. 113 channels were sampled at 1 m intervals over mineralised intersections. Standard Operating Procedures (SOP) include procedures for ensuring samples are representative.</td>
</tr>
<tr>
<td><strong>Drilling techniques</strong></td>
<td>DD (NQ for surface holes), RC (5.5 inch diameter holes) and PERC drilling; Channel sampling (hammered from twin sawn grooves).</td>
</tr>
<tr>
<td><strong>Drill sample recovery</strong></td>
<td>Drill sample recovery is not available in the database. Visual inspection of NRR drilled core suggests good recovery.</td>
</tr>
<tr>
<td><strong>Logging</strong></td>
<td>Logging was undertaken using the Kalahari logging codes, where material drilled, grain size and colour were described.</td>
</tr>
<tr>
<td><strong>Sub-sampling techniques and sample preparation</strong></td>
<td>Sampling was undertaken at every 1 m interval. This material was split using a riffle splitter to collect a 2 -2.5kg sample that would be sent for analysis. Samples were collected in sampling bags which were labelled on site. Orientation line to prevent preferential sampling of core is described in the SOP.</td>
</tr>
<tr>
<td><strong>Quality of assay data and laboratory tests</strong></td>
<td>Samples were sent to Genalysis Johannesburg for preparation and Genalysis Perth for analysis. Both Genalysis Laboratories are accredited according to ISO/IEC 17025. Further samples were sent to Bureau Veritas Swakopmund. Blanks and field duplicates were inserted in the sample stream on site. Field duplicates showed very good results, while Zn blanks showed high failure rate which indicates potential for contamination at the lab. No independent CRMs were inserted. Due to this check assays have been sent to Genalysis Johannesburg for re-assay.</td>
</tr>
<tr>
<td><strong>Verification of sampling and assaying</strong></td>
<td>5% of the previously analysed samples have been sent to Genalysis Johannesburg for re-assay.</td>
</tr>
<tr>
<td><strong>Location of data points</strong></td>
<td>The surface topography was surveyed and the collar positions of drillholes were also surveyed using a DGPS by African Geomatics. The profile and the collar positions agree with one another. Holes have been surveyed down hole with a reflex tool (DD) or Magsus (RC). A surveyed topography of the immediate mine area was provided by NRR.</td>
</tr>
<tr>
<td>Section 1: Sampling Techniques and Data Criteria</td>
<td>Explanation</td>
</tr>
<tr>
<td>------------------------------------------------</td>
<td>-------------</td>
</tr>
<tr>
<td>Data spacing and distribution</td>
<td>Holes in the Northern Extension were drilled on a 15m x 40m grid, with a range of 530 m along strike and 220 m across strike. Most samples were collected on a 1 m interval and the dataset was thus composited to 1 m. Data spacing in the South Mine varied. Additional drilling is required in some areas to increase the classification from Inferred to Indicated.</td>
</tr>
<tr>
<td>Orientation of data in relation to geological structure</td>
<td>All holes were drilled to intersect the orebody, though not all at 90°.</td>
</tr>
<tr>
<td>Sample security</td>
<td>Sample pulps are stored in a locked container on-site, where a site manager is based and resides on the premises.</td>
</tr>
<tr>
<td>Audits or reviews</td>
<td>Database audited, QA/QC report undertaken and site visit completed.</td>
</tr>
</tbody>
</table>
## Section 2 of the JORC Table 1 for the in-situ NLZP

<table>
<thead>
<tr>
<th>Section 2: Reporting of Exploration Results Criteria</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mineral tenement and land tenure status</td>
<td>NRR is a sole owner of EPL 2902 to the NLZP, which expires on the 17th April 2014. This is the last EPL issue to NRR.</td>
</tr>
<tr>
<td>Exploration done by other parties</td>
<td>Drilling on the in-situ Resource was done by ISCOR in the late 1980’s to early 1990’s and by Kalahari in 2008.</td>
</tr>
<tr>
<td>Geology</td>
<td>Reject material after processing is completed. It is pumped into the dam as slurry of fine grained material.</td>
</tr>
<tr>
<td>Drillhole information</td>
<td>164 holes were drilled (late 1980’s to 2013) and samples from 113 channels were collected (2011 to 2013) and analysed. All samples have been used in the estimation process.</td>
</tr>
<tr>
<td>Data aggregation method</td>
<td>All samples were collected at 1 m intervals and no aggregation of samples was undertaken.</td>
</tr>
<tr>
<td>Relationship between mineralisation widths and intercept lengths</td>
<td>All samples had equal length of 1 m. Mineralisation widths varied.</td>
</tr>
<tr>
<td>Diagrams</td>
<td>A plan view of drillhole collar locations is included in the report.</td>
</tr>
<tr>
<td>Balance reporting</td>
<td>Statistical assessment of grade distribution of the samples is included in the report.</td>
</tr>
<tr>
<td>Other substantive exploration data</td>
<td>Bulk density measurements were undertaken by NRR. The results averaged 3.45 t/m³ for in-situ bulk density.</td>
</tr>
<tr>
<td>Further work</td>
<td>Further work and infill diamond drilling have been planned for the purpose of assessing the in-situ Resource. Initial drilling has commenced during October 2013 and will continue until February 2014.</td>
</tr>
</tbody>
</table>
### Section 3 of the JORC Table 1 for the in-situ NLZP

<table>
<thead>
<tr>
<th>Section 3: Estimation and Reporting of Mineral Resources Criteria</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Database Integrity</td>
<td>Geological and sampling information is stored in a MS Access database. Data validation undertaken on the data did not show any significant discrepancies.</td>
</tr>
<tr>
<td>Site visit</td>
<td>The site visit was undertaken by Dr Belinda van Lente (Senior Consultant at Snowden; Pri.Sci.Nat) between 18th and 20th September 2013. The underground workings were visited, where the geology was reviewed and underground channel sampling witnessed. Furthermore, the sample storage facility where the remainder samples are being kept, including core, was also assessed. The database and storage was also discussed on-site. Dr Simon Dominy, the Resource CP, did not visit the site. He supervised Dr van Lente.</td>
</tr>
<tr>
<td>Geological Interpretation</td>
<td>The topography was modelled from the topographic survey undertaken by African Geomatics. The structure wireframes were modelled from surface measurements and projected downwards based on these measurements. The geology and mineralised envelopes were modelled based on drillhole data (grades and lithology). DHTEM plates and DD data was used for modelling in the Northern Extension. The South Mine interpretation was based on a DD and channel sample dataset, as well as previous stoping volumes.</td>
</tr>
<tr>
<td>Dimensions</td>
<td>The Northern Extension in-situ Resource vary in depth between 0 m and 180 m below surface. The ore lodes measure in total 580 m along the longest axis, in the north-west direction, and 275 m in total in the north-east, in its shortest axis and occupy a volume of 247,218 m³. The South Mine in-situ Resource vary in depth between 130 m and 240 m below surface. The ore lodes measure in total 250 m along the longest axis, in the north-west direction, and 60 m in total in the north-east, in its shortest axis and occupy a volume of 43,194 m³. The mineralised zones are thin pencil forms that plunge down dip.</td>
</tr>
<tr>
<td>Estimation and modelling techniques</td>
<td>Ordinary kriging was used, with modelled variograms for each element (Pb, Zn and Ag). The deposit was domained into ore and waste and only the ore portion was estimated. Any changes in dip or dip direction was taken into account by applying dynamic anisotropy, with searches employed in comparison to variogram ranges to limit the influence of samples that were far. Minimum and maximum numbers of samples used were 4 and 20, respectively. Though some correlations were established between Pb and Ag, each element was estimated individually. Slicing analysis, visual inspection and average comparisons between the model and composites were done. All three methods showed the estimates to be represented well by the composites.</td>
</tr>
<tr>
<td>Moisture</td>
<td>Moisture content was measured during the bulk density measurement studies.</td>
</tr>
</tbody>
</table>
### Section 3: Estimation and Reporting of Mineral Resources Criteria

<table>
<thead>
<tr>
<th><strong>Cut-off parameters</strong></th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cut-off parameters for extreme grades were applied based on histogram grade distributions. These were: 23.05% Pb, 33.48% Zn and 331 g/t Ag.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Mining factors and assumptions</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining will be via an underground mechanised operation utilising open stoping. It is expected that mining will be at a rate of 250,000 t per annum.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Metallurgical factors</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>The metallurgical factors are based on process and plant metallurgical recoveries determined by initial testwork. A conservative approach has been taken with respect to the ore metal recoveries and grades used to determine costs of production. Recoveries of Zn 85%, Pb 85% and Ag 70% are taken. Concentrate grades of 52% Zn and 55% Pb are taken.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Environmental factors</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>An EIA application will be made to the Government of Namibia. It seems to be the current policy of the Government to allow mining in game parks subject to the obtaining of relevant environmental clearances and rehabilitation obligations by the licence holders.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Bulk density</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Bulk density was provided by NRR as 3.45 t/m³ for in-situ bulk density. The density values for the in-situ ore from the database averaged at 3.50 t/m³.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Classification</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>The material was classified at Indicated and Inferred. The classification is based on the drillhole density, the number of samples and the search distance applied to estimate each block. Classification wireframes were created in plan view sections of 10 m intervals using CAE Datamine Studio 3™, and verified in vertical view sections of 15 m intervals. Areas were wireframed as Indicated where drillhole and channel sample spacing was generally within 15 m (X) by 40 m (Y) by 10 m (Z). Inferred wireframes were created for areas where the drillhole and channel sample grid spacing was at least 30 m (X) by 50 m (Y) by 15 m (Z). Additionally, for blocks to be classified as either Indicated or Inferred, a general geological continuity should be shown. This was determined by the variography and the search volumes calculated from the variogram ranges. Three search volumes were used, orientated along the strike, dip direction and the angle of dip of the orebody. Blocks that were estimated within the primary search volume, generally show geological continuity. Blocks estimated within the secondary search volume, cannot be classified as Indicated, but at the most as Inferred. Only ore blocks were classified, since the waste was not estimated.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th><strong>Audits or reviews</strong></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>This Resource estimate has been reviewed by the CP, Dr Dominy.</td>
<td></td>
</tr>
</tbody>
</table>
Appendix B  Competent Person’s Consent Form
Competent Person’s Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

North River Resources PLC:
Namibia Lead Zinc Project Study: In-situ Resource Estimate

Snowden Industry Mining Consultants Ltd

Namib Lead Zinc Project In-situ Resource

9th December 2013
Statement

I, Dr Simon Dominy, confirm that I am the Competent Person for the Report and:


- I am a Competent Person as defined by the JORC Code 2012 Edition, having over 24 years’ experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.

- I am a Fellow of The Australasian Institute of Mining and Metallurgy (205232) and the Australian Institute of Geoscientists (1921).

- I have reviewed the Report to which this Consent Statement applies.

I am a full time employee of

Snowden Industry Mining Consultants Ltd

and have been engaged by

North River Resources PLC

to prepare the documentation for

Namib Lead Zinc Project In-situ Resource

on which the Report is based, for the period ended

9th December 2013

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to the Mineral Resource.
Consent

I consent to the release of the Report and this Consent Statement by the directors of:

North River Resources PLC

Signature of Competent Person

[Signature]

Professional Membership: FAusIMM FAIG

Signature of Witness:

Date:

Membership Number:

Print Witness Name and Residence: (eg town/suburb)
Appendix C

NRR_EX_6.4017_RAB, Aircore, RC Drilling & Sampling
Purpose: To ensure that the integrity of all drilling data is maintained from source to database, in a format that is acceptable to all internationally accepted standards and to outline the requirements for maintaining a healthy work environment around dust producing drill rigs.

Benefit: RAB, aircore and RC drilling represent cost effective methods for exploration and resource definition. RAB and aircore drilling are usually carried out for geochemical sampling and are generally not of sufficient quality to be acceptable for use in resource estimations.

For all methods, however, incorrect drilling and sampling practices and/or poorly supervised surveys can lead to poor recoveries and contaminated or non-representative samples.

The successful application of RC drilling in exploration or resource definition programmes relies both on correct drilling procedures and adherence to correct sampling methods. Constant attention must be given to avoiding sample contamination and auditing must be employed to ensure that sampling is representative of the material drilled.

RAB, aircore and RC drilling all create a certain amount of airborne dust. To avoid inhalation of radioactive material – dust masks should be worn at all times when the rig is drilling and when sampling is occurring.

Scope: All personnel, at whatever level of involvement, need to be aware of the negative financial consequences and health risks that non-compliance to these procedures entail.

Remember
1. Always wear correct PPE.
2. Always minimize risk.
3. If you are unsure ask your supervisor.
4. Always tell someone where you are working.
5. Watch out for others and warn them of hazards.

PPE
Standard PPE consists of;
1. Leather steel capped boots.
2. Clear or tinted safety glasses
3. High visibility shirt, coveralls or vest
4. Hard Hat.

Procedure: RC Drilling

A face sampling bit should be used at all times for RC drilling.

Samples from RC rigs take time to reach the surface and it is important that the driller pauses at the end of each metre drilled and creates a break in the flow of sample to the surface. This is called “blowing back”. The bit is briefly...
lifted off the bottom of hole and the hole is blown clean. This system slows production slightly – but the driller should be instructed to carry it out to maintain sample quality. The sample bag should be removed from the bottom of the cyclone during blowing back.

If the first sample of each new rod is consistently showing a poor recovery, this may be due to water returning into the hole at each rod change – creating damp, sticky conditions. Damp samples will stick to the cyclone and other areas. This problem can be solved by drying the hole thoroughly with air following a rod change, before recommencing drilling and sampling.

RC drilling in areas of high water flows, creating wet samples is a potential area of serious sample contamination. Washing out and loss of the finer sample fraction may create unrepresentative samples by artificially concentrating or removing the mineralisation. The current practice for RC holes that are going to be used in future resource estimations, is if the drillers are unable to maintain dry samples – stop the hole. This is at the discretion of the supervising geologist, in discussion with the driller. If a full rod’s length (6m) has been drilled with wet samples it is probably time to stop the hole.

After completion of drilling, the hole should be blown clean and capped/plugged as soon as possible so that downhole surveying or re-entry by a diamond rig can be done at a later date.

**Sample layout**

The 1m bulk sample bags are laid out neatly in rows of 10 with the tops folded over (see Figure 1). Polyweave sacks containing the samples for submission to the laboratory and the reference calico bags should be neatly lined up so that the number of samples and IDs can be efficiently checked prior to removal from the drill site.

![Fig 1. 1m bulk sample bag laid out neatly in rows of 10](image)

**Geological Logging**

In conjunction with monitoring the sampling accuracy and quality, the geologist must carry out logging of the drill chips. A representative sample of the entire metre should be removed from the bag and not just a scoop off the top of the bag. The best way to achieve a representative sample of the metre is to use a strong PVC pipe which can be inserted into the chips for the entire depth of the bag. The PVC pipe should be cut at an angle to create a ‘spear’ for easier insertion into the chips. The chips are then removed from the PVC pipe into the sieve for washing and inspection by the geologist.
A small reference sample of chips from each metre is then placed in the chip tray marked with the hole ID and the metre intervals (see Figure 2).

The geologist should ensure that not only the chips in the tray are examined, but selections of the chips from the sieve using a hand-lens as much as possible. RC chips are difficult to describe in detail, but important information can still be described regarding lithology, texture, mineralogy, alteration and mineralisation despite the small samples.

Geological logging is carried out on the standard RC Logging Sheets (NRR_EX_6.1011) or directly onto portable field computers (e.g. iPAQ hand-held computers). The logging of RC chips has historically been done by the company on a metre by metre basis, although it is perfectly acceptable to log on an interval (unit) basis as well. It is often useful for geologists logging RC chips to examine drill core – if it exists – to assist in recognition of relevant rock units/alteration/mineralisation etc. There should be regular discussion between RC and diamond geologists to ensure, for example, that rock types are being logged consistently or that weathering/oxidation is being logged consistently across the deposit.

In the case of Uranium prospects, hand held spectrometer readings are taken by field technicians on every metre sample. Before readings are taken, a patch of ground where low background readings are obtained is located for taking the readings – so that results aren’t skewed by any outcropping mineralisation. The spectrometer is placed on the sample for 1 minute and the ppm value is recorded.

Figure 2. Washed RC chips in a chip tray ready for logging

**Sampling**

All sample bags (plastic and calico) should be clearly and accurately labelled in permanent marker pen. A system of sample books with removable tags is the preferred method where large volumes of sample are being generated. Prior to drilling a hole the geologist prints out a sheet detailing the sample intervals and sample numbers for that hole. The removable tag, bearing the
sample ID, is attached to the plastic sample bag used for submission to the laboratory for analysis (see Figure 3). The stub left behind in the sample book is filled out with the hole and interval details for the sample, ensuring that there is a permanent record kept that can be referred back to - if needed at a later date (see Figure 3).

Sampling accuracy of the drillhole, metre interval and sample ID information is monitored by the RC site supervisor and the rig geologist on a regular basis throughout the drilling of a hole. It is often difficult or impossible to correct errors in sampling after the fact, so special attention is needed to sampling of RC drillholes.

Figure 3. (Left) 2Kg sample for lab submission; (Right) Sample ticket with hole ID and from – to information

QA/QC check samples in the form of standards, field duplicates and blank samples are inserted into the sampling stream i.e their sample IDs form part of the general sample sequence. Standards, blanks and field duplicates are inserted at a rate of approximately 1 in every 20 samples. These QA/QC samples should not be inserted in exactly the same order or sequence and should be varied.

Sampling of the 1m bulk samples is carried out by splitting through a 3-tier riffle splitter – creating an approximately 4kg sub-sample. This sub-sample is then split, into 2 equal 2kg portions, by pouring the sample over the full width of the lower tier of the splitter. One of the 2kg portions is placed in a plastic sample bag and it is tied shut with a cable tie – this is the sample for submission to the laboratory. The other portion is stored as a reference/archive sample.

In the case of exploration RC or RAB drilling – 5m composites are sometimes taken in zones where significant mineralisation isn’t expected to be found e.g. alluvial cover. These samples are taken using a spear. Each metre sample should be homogenised, by holding the bag shut and kneading or massaging the bag and the spear should be inserted through the full depth of the sample.
In the case of wet sampling, to avoid contamination of the splitter by wet material, these samples are manually sampled by hand. The bulk 1m sample bag is kneaded and massaged to homogenise the sample as much as possible and then handfuls of sample are removed from throughout the depth of the sample to create an approximately 2kg sample.

Contamination of the splitter should be kept to a minimum and should be regularly checked to ensure that material isn’t building up on the riffles or chutes. During sampling, banging on the splitter with a rubber mallet is usually sufficient to remove any lodged material. Sometimes it may be necessary to borrow a compressed air line from the drillers to remove any sticky or hard to get at material. Wet samples shouldn’t be tipped through the splitter.

Samples being split using the riffle splitter should be evenly poured across the full width of the riffles.

It is important that the cyclone remains clean and free of built up material. The cyclone should be cleaned, by the drillers, as a matter of course at the end of each hole. If the geologist suspects that material may be building up in the cyclone, if wet or damp intervals have been intersected, then he/she should request that the drillers clean the cyclone at the next rod change.

**Recoveries**

Ideally, recovery of samples should be monitored by weighing each individual bulk sample bag. In the case of dry samples, this weight can be used to estimate a recovery percentage for each metre – using the hole diameter and estimates of bulk densities to calculate the in-situ sample weight. Alternatively a more qualitative system can be used: e.g. poor (less than 1/3 of a typical sample); moderate (1/3 to 2/3 of a typical sample) or good (greater than 2/3 of a typical sample).

---

**Related Documents**

- NRR_EX_6.1011_RC Logging Sheet
- NRR_EX_6.1011_RC Sampling Sheet (within same Excel file)

Approved: ___________________________ Dated: ___/___/___
Appendix D  NRR_EX_6.4021_Chip Channel & Channel Sampling
Purpose: The purpose of the Chip Channel and Channel Sampling Procedure is to instruct on the procedure for safely and correctly sampling mineralized outcrop over specific intervals.

Benefit: Channel sampling can be a useful early stage and advanced stage exploration tool, but must be carried out properly to be effective.

Channel sampling can be a safety hazard, both from splintering rock chips and use of rock cutting tools; special attention to safety is required.

Scope: To ensure that chip channel & channel sampling is standardized within the company. Channel sampling is like horizontal drilling and provides important information regarding grades. The longer the interval the better the estimate of grade continuity will be.

Remember
1. Always wear correct PPE.
2. Always minimize risk.
3. If you are unsure ask your supervisor.
4. Always tell someone where you are working.
5. Watch out for others and warn them of hazards.

PPE
Standard PPE consists of;
1. Leather steel capped boots.
2. Clear or tinted safety glasses
3. High visibility shirt, coveralls or vest
4. Hard Hat.

Procedure:
1. Safety glasses are essential PPE for rock sampling.
2. Field equipment required: Geological hammer, chisel, notebook, hand lens, GPS, scraper, permanent marker, buff tape, sample bags, tape measure.
3. If mineralized outcrop is exposed over a significant strike length a much more meaningful method of sampling is by means of chip channel/channel sampling (perpendicular to strike) as opposed to rock chip sampling. If results from this sampling suggest good potential for an economic deposit more accurate, mechanized channel sampling can be employed.
4. The methods employed for both types of channel sampling are similar and are outlined below:
   - Clean the mineralized outcrop by removing debris and dirt.
   - Measure the outcrop in meters using a tape measure. Leave the tape measure on the ground so it is clearly visible and mark off the intervals. For highly variable mineralisation use 1 metre samples and for more uniform mineralisation a 2 metre sample interval can be used.
   - Note the date and channel start point coordinates in your
field note book.

- Study the outcrop, map the geological features and sketch the interval with geological features in your notebook. Remember to add the scale and outcrop direction (For Example east-west or an arrow indicating 090 direction) in your drawing as well as the sample localities. Geological data will be entered into the Channel Sampling Records Form as if this were a horizontal borehole (NRR_EX_6.1009).

- The outcrop can be sampled in several ways dependant on the stage of the project and the nature of the outcrop.
  - In the early stages of a project if the outcrop is sufficiently soft, or outcrops in such a way that a continuous linear channel sample, (perpendicular to strike) can be collected with a chisel, then this is the preferred method.
  - If the outcrop is too hard to collect a continuous channel sample with a chisel then divide the outcrop into 1 square meter panels and conduct chip channel sampling. Make sure the 1 meter square panels are representative of the outcrop and collect a total of 1 – 2Kg of material from different areas within each square.
  - At a more advanced stage of the project a double headed diamond blade grinder can be used to cut a slot that is easily chiselled out and forms a continuous channel sample.

- Note the rock characteristics, vein density / sulphide-oxide percentages and mineral assemblage for each sample interval in your field note book. Note degree of weathering of your outcrop.

- Remember to take a GPS reading at the location for the last sample so that there is a start and end point to all channel samples.

5. Sample IDs must be marked in capital letters in the following manner:

- Identify the Project/License within which the Channel sample is found. For example, Ubib Concession will use **UB**.
- Identify the type of sample, a chip channel sample is marked with **CC** and a chiselled or mechanised channel sample is marked with **CH**.
- Mark the number of the channel being sampled **001**. This number should be unique within the project or prospect area to avoid duplication and confusion (i.e. just like any borehole numbers within a project or prospect area should be unique).
- Mark the interval -01, -02, etc for each sampled interval
- So, the first chip channel sample of the 10th channel line within in the Ubib Project would be submitted for assaying with the following label: **UBCC010-01**, similarly the 10th sample along the 4th mechanized channel sample line at Tsawisis Prospect at Ubib would be numbered **TWCH87-10**.

6. Samples should be packaged with a loose sample card in the bag and a second folded into the hem of the sample bag. Samples are then submitted for assay as per the Shipping
### Samples for Assay Procedure.

7. Copies of field maps showing channel sample locations should be made and kept in the Windhoek office. Data should also be recorded electronically using the borehole logging forms and stored in the relevant folder on the server in Windhoek.

### Related Documents

- NRR_EX_6.4002_Health & Safety
- NRR_EX_6.4007_Vehicles
- NRR_EX_6.4006_Traversing
- NRR_EX_6.4007_Environmental Considerations
- NRR_EX_6.4012_Shipping of Samples for Analyses
- NRR_EX_6.1011_RC Logging Sheet

**Approved: ___________________________ Dated: ___/___/___**
Appendix E  NRR_EX_6.4022_Bulk_Density
Determinations Core
# Purpose
To describe the correct procedure for calculating bulk density of rock types from core samples.

# Benefit
Bulk density measurements of rock types are critical to any resource or reserve calculation and should be measured as a matter of course as part of any diamond drilling program.

# Scope
To define the requirements for carrying out bulk density measurements on core samples.

# Remember
1. Always wear correct PPE.
2. Always minimize risk.
3. If you are unsure ask your supervisor.
4. Always tell someone where you are working.
5. Watch out for others and warn them of hazards.

## PPE
Standard PPE consists of:
1. Leather steel capped boots.
2. Clear or tinted safety glasses
3. High visibility shirt, coveralls or vest
4. Hard Hat.

## Procedure: Specific Gravity determination of Drill Core:

### Measuring and Quality Control Procedure

Scope: This procedure is suitable for the determination of bulk density or specific gravity of solid objects, such as drill core and rocks.

Principle: The specific gravity of a non-porous to minimally porous solids can be determined from its loss in weight when weighed in water.

Apparatus:
1. Top loading balance, sensitive to 0.1g and fitted with a built in under hook. The balance should have a capacity of at least 4000g.
2. 30 small buckets for the overnight soaking of samples.
3. Place the balance on a table with a small hole so that the under hook can be accessed from underneath.
4. Secure a light weight harness (fine wire connecting a fine wire basket) to the under hook.
5. Totally immerse the sample and harness in a large bucket (plastic rubbish bin) of water without it touching the sides or bottom.
Measuring procedure for each sample:

1. Measure the dry weight of each sample and record the weight (W1).
2. Immerse the first thirty samples in separate buckets of water overnight, with the details of each drill core sample (interval, rock type, etc.) written on the front of the bucket.
3. Measure the weight of each soaked sample and record the weight (W2).
4. Minus the weight of the soaked sample (W2) by the weight of the dry sample (W1) to get the weight of contained water (W3).
5. Secure a light weight harness to the under hook and totally immerse the harness in the large bucket of water without it touching the sides or bottom.
6. Reset zero on balance.
7. Place the soaked sample in the submerged harness, so that it is also totally immersed.
8. Measure the weight of each sample and the harness in the water and record the weight (W4).
9. Assess if the weight of contained water (the porosity) is large enough to significantly alter the specific gravity calculations.
10. If the weight of contained water has a significant effect on the specific gravity calculation then continue with the overnight soaking procedure. However, if the weight of contained water was insignificant enough to affect the specific gravity calculation (e.g. very low porosity) then the overnight soaking procedure may be skipped.

Quality control of measuring procedure:

1. Repeat the measuring procedure for the first ten samples in order to establish the repeatability of results.
2. If satisfactory, repeat the measuring procedure for every tenth sample in order to establish that the satisfactory repeatability of results is continuing.
Calculation:

\[
\text{Dry bulk density} = \frac{W1}{W1 - (W4 - W3)}
\]

Sampling Technique:

1. Determine the Dry Bulk Density of all mineralised drill core samples sent for assay, using the half core that has been left in the core trays.
2. Measure representative samples of the various rock packages: Note that all rock types should be included – not just the mineralised rock units; the bulk density is required for all rock types that would be mined, including waste rock.
3. As a general rule Bulk Density determinations should be done every 5m in a drill hole, ensuring a good spread of samples throughout the hole (oxidised and un-oxidised samples).
4. Collect enough samples from all the drill holes until a significant database of reasonable Dry Bulk Density determinations are accumulated for each rock type.

Related Documents

NRR_EX_6.1012_Bulk Density Measurement Form

Approved: ___________________________ Dated: ___/___/___
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1. SUMMARY

1.1 Introduction

North River Resources is currently developing its Namib North Mine (NNM) zinc project located in southern Namibia near the border with South Africa. The mine is located adjacent to the town of Rosh Pinah at about 400 m above sea level. With an initial mine depth of about 200 m, heat and heat stress are not expected to cause any problems that require any action other than appropriate ventilation.

The study is to provide a review of the ventilation requirements and ventilation arrangements initially for mining down to 200m below surface. This includes identifying applicable design criteria leading to suitable ventilation standards. The project is an underground mine with shrinkage stoping and a production rate of 20 000 t/month. The effects of an increase in mine depth will also be considered.

1.2 Findings

1. The design criteria are set out in section 2 with the background given in Appendix A1. Allowable gas concentrations are those current in South Africa and generally the same as the American Conference of Governmental Industrial Hygienists (ACGIH) with no reductions in exposure concentration for extended shift lengths.

2. In respect of respirable quartz, although a standard of 0.2 mg/m³ has demonstrated to eliminate silicosis, most jurisdictions use a value of 0.1 mg/m³ and this standard is proposed for NNM. There is, however, a very low or negligible quartz content of the host rock and a limiting respirable dust concentration of 5.0 mg/m³ is proposed.

3. The recommended design diesel exhaust ventilation rate is 0.05 m³/s per kW and this should ensure that ambient gas concentrations are less than one third of the standards. The proposed diesel particulate matter (DPM) standard is the same as that recently introduced in Ontario, Canada at 0.4 mg/m³.

4. The indicated mine ventilation rate for a production rate of 20 000 tonnes per month is 125 m³/s. A mine ventilation rate of 150 m³/s was confirmed by an examination of the ventilation standards for different activities. The ventilation rate includes an allocation for the decline haulage using diesel powered trucks.

5. Development ventilation, particularly for long declines from surface was specifically examined and the use of 100 m lengths of 915 mm diameter low leakage duct with a 45 kW fan is recommended to control leakage, provide adequate diesel exhaust dilution and reasonable re-entry times after blasting.

6. For the short developments the fans required using 760 mm duct are 15 kW. Alternatively, compressed air fans could be used such as the Korfmann DV6. The fan has a compressed air consumption of 0.07 m³/s or about 40% more than a standard rock drill used with a jack leg. Although a separate electrical supply is not required, the fan efficiency is at best about one sixth that of electric powered auxiliary fans.

7. Optimum airway sizes were considered for lateral airways (including declines), raise bore holes and long hole raises with the optimum air velocities being 8.0 m/s, 15.5 m/s and 6.0 m/s respectively. For haulage declines, the optimum velocity is greater than
that normally used to control dust and additional dust mitigation measures such as water sprays may be necessary.

8. The recommended overall ventilation arrangement is to use the haulage declines and the surface shaft as intake airways. Ventilation air flows both north and south on the sublevels to the ore shoots and back to Junction stope for exhaust to surface.

9. For intake between surface and the working areas, it is suggested that two declines should be used in parallel with the surface shaft. Haulage decline sizes are assumed to be 3.2 m x 3.5 m and, from the ventilation point of view, do not need to be larger.

10. The mined out Junction stope is the main return to surface and connected to the stoping areas by existing (collection) drives. The connection through the crown pillar to surface should be enlarged to 18 m² (3.0 m x 6.0 m).

11. A feature of shrinkage stoping is the variable cross sectional area for airflow caused by blasting breasts/benches followed by draw down of the “swell”. The contractions and expansions created cause shock losses leading to higher pressure losses. Assuming that the velocity is limited to a maximum of 4 m/s (determined by the main fan pressure available), the pressure losses in each contraction and expansion is about 10 Pa.

12. Detailed ventilation network simulations are not necessary at this stage of the design with 90% to 95% of the pressure losses in the main intake and exhaust airways. For the recommended ventilation arrangement, the maximum main fan pressures are about 550 Pa and the total main fan power required is about 120 kW.

13. With a limited mine life of less than 10 years, a relative low pressure system and reasonable power costs, the main fans do not have to be designed for a long life and a high efficiency. The duty is suitable for axial flow fans and this can be used to reduce overall costs by selecting a much simpler inlet box arrangement.

14. The estimated cost of the main fan installation on surface is about US$ 310 000 and this is to be confirmed as soon as a budget price is obtained. The operating fan power and maintenance costs would be approximately US$ 150 000 per year.

15. Auxiliary fans are required for both long and short development. The overall auxiliary fan power is estimated to be less than 150 kW with a capital cost of about US$75 000. The total annual operating costs (power, maintenance and repair) is also estimated to be US$ 150 000.
2 VENTILATION REQUIREMENTS

2.1 Ventilation design criteria

The background to the design criteria used in this review is provided in Appendix 1 and the recommendations are summarised as:

Air velocity - target design minimum 0.5 m/s.
   Basis – good practice and control of heat stress.

Dust - Average respirable dust concentration less than 5.0 mg/m³.
   Minimum 95% compliance with 10.0 mg/m³ of respirable dust.
   Pro rata reduction in standards for extended shift lengths.
   Basis – International Standards and good practice

Gases - Carbon monoxide (CO) TWA = 50 ppm STEL = 100 ppm
   - Carbon dioxide (CO₂) TWA = 0.5% STEL = 3%
   - Nitric oxide (NO) TWA = 25 ppm STEL = 35 ppm
   - Nitrogen dioxide (NO₂) TWA = 3 ppm STEL = 5 ppm
   - Sulphur dioxide (SO₂) TWA = 2 ppm STEL = 5 ppm
   Basis – South African and International Exposure Standards

Notes
1. TWA is Time Weighted Average and STEL is Short Term Exposure Limit
2. No modification is recommended for shifts shorter or longer than 8 hours for any of the above gases.

Diesels - A minimum air dilution rate of 0.05 m³/s per kW of power at the point of use and air leakage is taken into account separately.
   - minimum 95% compliance with 0.4 mg/m³ diesel particulate matter (DPM)
   Basis - International mining standards (Ontario, Canada) and good practice.

   - 30.5°C - 32.5°C, Modified work status, re-location or imposed work/rest regimen depending on the wet bulb temperature
   - 32.5°C, Stop work until corrected
   Basis – Mining and Tunnelling Codes of Practice

Noise - 85 dBA continuous exposure
   - 100 dBA intermittent exposure
   Basis – International standards and good practice

2.2 Climate and heat

The climate in the part of south western Namibia where the mine is located is arid with less than 100 mm annual rainfall. Summer maximum dry bulb temperatures reach the mid 30’s however the relative humidity is low at between 30% and 35%. The indicated wet bulb temperatures on the warmest days would then be between 21.5°C and 22.5°C and this range could be considered the 2.5% design values (wet bulb temperatures only exceeded for about 70 hours/year during the summer). The mine elevation is approximately 400 m above sea level and the design barometric pressure is 97.0 kPa.

The rock temperature on surface (actually measured about 50 m below surface to eliminate seasonal variations) is normally equal to average annual dry bulb temperature and this is about 18°C. The highest geothermal gradients encountered in mining are between 25°C and 35°C
per km of depth. It is therefore unlikely that the virgin rock temperature at a depth of 200 m will exceed 25°C (18 + 0.2 x 35). Heat flow from the surrounding rock or any ingress of fissure water is therefore unlikely to be significant and can be ignored.

The remaining heat loads are from auto-compression (Joule-Thompson effect) and mine equipment. Auto-compression results in approximately 0.5°C wet bulb temperature increase for each 100 m depth. Where diesel powered equipment operates, the increase in wet bulb temperature in the working places are modified by the thermal flywheel effect of the surrounding rock and typically average about 3°C. This value is relatively constant irrespective of the amount and power of the equipment used and is mainly a function of the ventilation rates.

Other equipment heat loads are associated with the use of electric power such as fans and pumps. These are expected to be quite small and mostly offset by the cooling effect of expanding compressed air providing power in drills and fans (not venturis). The overall wet bulb temperature increase is likely to be highest where diesel equipment operates particularly in development where increases up to 4°C could be expected.

The expected maximum underground wet bulb temperatures are therefore between 25.5°C and 26.5°C and, providing that a minimum air velocity of 0.5 m/s is maintained, heat stress conditions should not be a problem. Although the highest wet bulb temperatures are associated with the operation of diesel powered equipment, it should not be necessary to provide the equipment with air conditioned cabs.

### 2.3 Indicative mine air quantities

#### General comments

In the absence of detailed mining plans and schedules, a table of baseline values can be used to provide an estimate of the ventilation requirements based on the mining method and rock handling system. Although general design air quantities are not appropriate where detailed mine and ventilation design is available or possible, they are supportive of the criteria being used for design and significant deviations from normal values need to be explained and justified. Generally:

\[
\text{Mine quantity} = \alpha t + \beta
\]

Where \( t \) is the annual production in million tonnes per annum (Mtpa), \( \alpha \) is a variable air quantity factor, which is directly related to production rate and the mining method and \( \beta \) is the constant air quantity required to ventilate the mine infrastructure such as the ore handling system. Typical values of the variable air quantity factor \( \alpha \) for different mining methods are given in the following table.

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Air quantity factor (m³/s/Mtpa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Block caving</td>
<td>50</td>
</tr>
<tr>
<td>Room and pillar (continuous miner)</td>
<td>75</td>
</tr>
<tr>
<td>Sub level caving</td>
<td>125</td>
</tr>
<tr>
<td>Sub level open stopes &gt; .25 Mt</td>
<td>175</td>
</tr>
<tr>
<td>Sub level open stopes &lt; .25 Mt</td>
<td>250</td>
</tr>
<tr>
<td>Room and pillar (conventional)</td>
<td>225</td>
</tr>
<tr>
<td>Mechanised cut and fill</td>
<td>325</td>
</tr>
<tr>
<td>Non-mechanised mining</td>
<td>400</td>
</tr>
</tbody>
</table>
The constant air quantity $\beta$ is mainly dependent on the ore handling system and, to a certain extent, the overall mine production rate. For mines where part or all of the broken rock is hauled to surface using diesel powered trucks, a suitable value of $\beta$ for the decline haulage can be obtained using $7.5 \text{ m}^3/\text{s}$ per Mtpa.km. This value is reasonably independent of the size and power of the trucks used with higher powered trucks usually operating at an increased speed and having a shorter cycle time as indicated in Appendix 2. A minimum haulage ventilation rate of $25 \text{ m}^3/\text{s}$ is applied in shallow mines, low production rates.

For a mine using shaft or conveyor hoisting where the ore is collected in ore passes and crushed underground, a typical value of $\beta$ is between $75 \text{ m}^3/\text{s}$ to $100 \text{ m}^3/\text{s}$ and depends on the crushing and ore pass arrangements. As the ore handling systems become more extensive i.e. using additional conveyors or other ore transfer systems, $\beta$ can increase by up to 50%. On very large mines where multiple shaft systems are used, the constant air quantity $\beta$ is also a multiple of the number of shaft systems required.

**Specific values for Namib North mine**

The mining method is shrinkage stoping with no back fill. From Table 2.1 the specific air quantity for non-mechanised stoping is $400 \text{ m}^3/\text{s}$ per Mtpa. The planned rock handling system comprises diesel powered loaders from stope draw points to truck loading points and diesel truck haulage to surface through a 1:8 decline. The truck haulage from 200 m below surface to surface based on a production of 0.24 Mtpa ($20,000 \times 12/10^6$) is approximately 0.4 Mtpa.km.

The indicated ventilation rate for the shrinkage stope mining is $96 \text{ m}^3/\text{s}$ (0.24 x 400) and can be rounded up to $100 \text{ m}^3/\text{s}$. At the required truck haulage ventilation rate $\beta$ has a value of $3 \text{ m}^3/\text{s}$ (0.4 x 7.5) and a minimum of $25 \text{ m}^3/\text{s}$ is applied. The indicated overall ventilation requirement for Namib North is therefore $125 \text{ m}^3/\text{s}$ ($100 + 25$).
3 VENTILATION REQUIREMENTS BASED ON MINING ACTIVITY

The design criteria summarized in Section 2.1 are used to determine the ventilation requirements and the indicative values given in Section 2.3 are generally based on these criteria. An alternative approach to using indicative values is to consider the ventilation of individual activities and to apply to the number of activities required to achieve a given production rate.

3.1 Development ventilation

Re-entry criteria

Long development headings are mainly associated with the mine infrastructure such as main access declines and exploration drives. Long developments are usually more than 250 m long before through ventilation can be provided and the maximum length of heading is only limited by the practicalities of auxiliary ventilation systems.

With respect to re-entry time after blasting, the following relationship can be used to determine the air quantity required to meet a specific re-entry period.

\[
Q = \frac{AL}{2t} \left[ 1 - \frac{18.5 \sqrt{A}}{L} \right] + \ln \frac{1000 F_a}{L} + 1
\]

Where:
- \( Q \) = air quantity for specific re-entry period (m³/s)
- \( A \) = cross sectional area (m²)
- \( L \) = length of heading (m)
- \( t \) = required re-entry period (s)
- \( F_a \) = face advance each blast (m)

The air quantities determined are based on averaging the re-entry times assuming perfect mixing of the blast gases with the ventilation air and those assuming no mixing at all where the blasting fumes would move out of the heading as a ‘plug’. In practice there is some diffusion and turbulent mixing at the fresh air/blast gas interface and the blast gases would tend to move out of the heading as an expanding plug hence the averaging of the two values.

The design air quantities are also based on the amount of air delivered to the face and, with duct leakage; the amount delivered to the heading is greater by an amount depending on duct leakage and the type of fan used.

Diesel exhaust dilution

Minimum diesel exhaust dilution requirements are really duplicating the requirements for ambient gas and diesel particulate contaminant levels. Providing that the contaminant levels meet the standards, a minimum dilution requirement is of limited relevance. Gas and particulate measurements are now relatively simple to take and can be accurately assessed which was not necessarily the case when large mobile diesel powered equipment was first introduced into mines about 45 years ago and consequently, minimum diesel exhaust dilution rates then had a greater relevance.

As indicated in Appendix 1 (Section A1.2) the fuel quality usually determines exhaust dilution requirements and often the amount of sulphur is a main determinant. Maximum levels of carbon monoxide and oxides of nitrogen in the undiluted diesel exhaust are usually specified as part of the approval to use diesel powered equipment underground and are typically 1500 ppm for CO and 1000 ppm for NOₓ.

An exhaust dilution rate of 0.05 m³/s per kW is recommended for Namib North mine and is
consistent with the dilution rates used in many countries. The Ontario statutory value of 0.06 m$^3$/s per kW was introduced in the early 1990’s and to date there is no definitive evidence of this higher value being based on risk. As explained in section A1.2, a dilution rate as low as 0.04 m$^3$/s per kW has been demonstrated to control diesel emissions to about one third of the normally accepted gas and particulate standards given in Section 2.1 above.

The actual low (one third) exposure values indicated above even with low exhaust dilution rates are a consequence of the lower actual utilisation of the diesel powered equipment. The average engine load of a loader undertaking a typical load-haul-dump cycle has been measured at between 65% and 70% of the full load power. This is similar to a haul truck where it will be close to full load when hauling up the decline but has significant periods of either idling or low engine load when returning down the decline.

The actual utilisation as a proportion of the time at full engine load is between 30% and 40% for loaders and trucks, 10% to 15% for service equipment and 2% to 5% for light vehicles. These have been measured and confirmed with fuel consumption data. The exposure values used as standards are time weighted averages and excursions are permitted providing that the time weighted average is not exceeded.

For these reasons it is acceptable to use the loader or loader and truck rated diesel engine powers only to assess the diesel exhaust dilution requirement for a development heading and to ignore the contribution from other equipment such as service vehicles. As a consequence of the cycle times, it is most unlikely that there will be more than one truck and one loader at close to full engine power at any one time in a heading up to 1000 m long.

**Duct systems**

All ducts used in auxiliary ventilation systems used to ventilate development headings leak, the extent of which depends on the type of ducting and its condition (i.e. the number and size of "holes") and the pressure difference between the air inside and outside the duct. This pressure difference is highest at the fan and lowest at the duct discharge and causes non-uniform leakage airflow along the duct even with a uniform "hole" size.

The three main types of duct used for auxiliary ventilation systems are; standard flexible, low leakage flexible and rigid steel duct. Standard ventilation duct is usually a single or double coated woven polyethylene fabric with a weight of between 200 g/m$^2$ and 300 g/m$^2$ and has either sewn or welded seams. Double ring or Velcro couplings would normally be used and the duct supplied in 15 m or 20 m lengths.

Low leakage duct has a polyester scrim and a thick PVC coating having a weight of between 600 g/m$^2$ and 800 g/m$^2$. The duct has welded seams and usually a minimum working pressure of 12 kPa. Double ring and toggle clamp or zipper couplings provide the minimum standard required. The cost of low leakage duct is approximately double that of standard duct and it is normally supplied in 50 m or 100 m lengths.

When using low leakage duct, standard duct in shorter lengths would be installed initially and, when at least 140 m (seven 20 m lengths) has been installed, the 100 m (five 20 m) lengths of standard duct closest to the fan would be replaced with low leakage duct.

Steel duct is normally spiral wound on site from 1.2 mm thick (18 gauge) galvanised strip and, for 762 mm duct, has a mass of 30 kg/m or about 180 kg for a 6.0 m length including angle section flanges. The couplings are usually also angle section and have a foam rubber insert to reduce leakage and to allow some flexibility in the duct alignment. This usually only results in about a 275 m radius for 6.0 m duct and, if a decline has “tighter” turns, additional steel “bend” sections are required.
**Duct leakage**

Leakage in new ducts, both flexible and rigid, should be limited to the imperfections in the couplings and any seals when connecting two lengths together. A reasonable simplification is to consider the leakage over 375 m and assess longer lengths as a power function. For example, for new standard flexible duct with couplings at 20 m, the leakage is 40% i.e. the supply at the fan must be 1.40 times that required at the fan. At a distance of 750 m, the leakage factor would be 1.40^2 or 1.96 i.e. 96% more air must be supplied.

This is a practical design value when using new standard flexible ventilation ducts with Velcro straps or similar connectors such as wire and grommet locators and internal overlaps to minimise leakage.

If the couplings are either zippers or spring steel rings sewn and welded into the ends of the duct and a flexible channel shaped toggle strap fitted over the rings and clamping them together, the leakage would be halved to about 20%. Such couplings would be used for low leakage flexible duct and, if the duct lengths were increased to 100 m with the same couplings, the leakage factor for new duct would decrease to about 4%.

For steel ducts, providing that the couplings are installed correctly, the leakage for the 6 m lengths would be about 2% and double this for 3 m lengths.

Once duct is installed in a developing heading it is subject to damage resulting from both blasting and contact with mobile equipment. Blast damage is minimised by advancing the three or four duct lengths closest to the face with the face and installing new ducting out of the main blast damage zone. Contact damage from mobile equipment is a function of the standard of duct installation and the clearance between the loaded trucks and the ducting.

With reasonable care during installation, duct replacement to minimise blast damage and using minimum duct clearances of 0.3 m, the duct leakage factors should not worsen to about double the new duct leakage. For standard or re-used low leakage duct these values should be doubled again. This presumes that any damaged duct is repaired by sewing tears and covering the repair (on the inside of the duct) with an adhesive patch.

**Fan and duct systems for development**

There are many duct and fan arrangements possible to ventilate development headings including multiple ducts for larger headings. Although costs are site specific, the overall cost (capital and operating) normally varies between 8.0 and 11.0 US$/m³/s/m for low leakage duct and between 11.0 and 14.0 US$/m³/s/m for standard flexible duct and rigid (steel) duct. It is normally impractical to use standard flexible duct for distances greater than 750 m.

With respect to diesel exhaust gas dilution, the size and length of the heading tends to determine the amount of diesel power required for both loaders and trucks. Typically, approximately 20 kW/m² of cross sectional area of diesel power is required to achieve reasonable face advance rates. Based on a diesel exhaust dilution rate of 0.05 m³/s per kW of rated power, the minimum ventilation rates are 1.0 m³/s/m² of cross sectional area.

**Long development**

Long development headings are mainly associated with the mine infrastructure such as main declines and exploration drives. These generally are between 375 m and 750 m long and would require sufficient ventilation for both a loader and a truck. For a 3.2 m by 3.5 m heading, the face ventilation requirement would be 11.2 m³/s (11.2 x 20 x 0.05).

If standard duct was to be used, leakage could vary between 40% and 100% and twin 760 mm
ducts or a single 1070 mm duct would be the minimum required. The airflow necessary to avoid
re-circulation at the fan location is about 20 m$^3$/s. Low leakage duct can also be used and a
single 915 mm duct would then be suitable to supply 13 m$^3$/s to the heading with a 15% to 20%
leakage rate. The airflow necessary to avoid re-circulation at the fan location is about 15 m$^3$/s
and this is the allocation for primary or “long” development.

Re-entry periods would be between 10 minutes and 20 minutes and only exceed 30 minutes if
the heading length is longer than 1000 m. This is mainly because duct leakage increases the
heading ventilation rate.

**Short development**

Short or most stope development can be defined as having only the loader entering the
auxiliary ventilated heading or stope. The truck used for rock removal stays outside the
heading in through ventilation. The limitation on ventilated heading length with auxiliary
ventilation systems is determined by the economics of load-haul-dump rock removal and is
usually about 250 m. In this type of short development it is normal for a development crew
to have multiple headings available.

Based on a 80 kW loader being the only diesel powered equipment entering a heading, the
minimum air requirement for diesel exhaust dilution at 0.05 m$^3$/s per kW is 4 m$^3$/s. For this
face delivery air quantity, the face air velocity would be about 0.35 m/s for a heading cross
sectional area of about 11 m$^2$ which does not meet the design criteria for minimum air
velocity. The design quantity should be increased to 6 m$^3$/s and this also easily meets the re-
entry criterion which would be about 10 minutes for a heading length of 150 m.

With respect to duct leakage, for standard flexible duct in 20 m lengths, the leakage factor
over an average distance of 150 m is between 25% and 30% resulting in a fan delivery
quantity of 7.0 m$^3$/s through a 760 mm diameter duct.

Each crew involved with short development would probably use one loader, jack leg drills
etc. (the truck loading point would be in through ventilation) and it would not necessary to
provide sufficient intake capacity for 7 m$^3$/s in each of the multiple headings available for
development i.e. 21 m$^3$/s for three headings.

A reducing fresh intake air quantity depending on the number of headings available has been
found to be suitable in most applications. The suggested ventilation allocations are 100% for
the first heading, 75% for the second, 50% for the third, 25% for the fourth and no additional
air for the subsequent headings.

If the design ventilation for one heading is 7 m$^3$/s, the maximum allocation for three
headings is 225% or about 15 m$^3$/s. This may be considered as the ventilation allocation for
each development crew undertaking stope or “short” development.

**Summary ventilation requirements for development**

Based on the information given above and the use of two development crews, one for
primary development and one for stope development, the overall ventilation requirement for
development is 30 m$^3$/s (15 + 15).

3.2 Ventilation for mining and rock removal

**Mining**

To achieve a 20 000 tpm production rate from shrinkage stoping where individual stopes
have a productivity of between 1500 to 2000 tpm (average of 1750 tpm) requires about 12
working places. This stope production rate is achieved by breaking (drilling and blasting)
about 25 m$^3$ of rock in each stope each 24 hours. Allowing a 25% variation for stopes in preparation or closing down results in 15 shrinkage stopes requiring ventilation.

The ventilation allocated for each stope is determined mainly from the minimum air velocity requirement of 0.5 m/s. Assuming that the largest stope opening is 16 m$^2$, the design airflow rate is 8 m$^3$/s. If the stope opening is larger than 16 m$^2$, an air mover would be necessary to locally increase the air velocity where personnel are working.

Where working places are in series with others in the same ore zone, a higher air quantity through the stopes could be used with the overall airflow restricted by the main fan pressure. The main restriction would be a maximum air velocity where dust becoming airborne would become a problem. This is usually limited to 6 m/s in main airways such as declines however a limit of 4 m/s would be more practical in stopes.

Overall, 15 stoping activities need to be ventilated and, based on 8 m$^3$/s per stope, the overall ventilation requirement for mining is 120 m$^3$/s (15 x 8).

**Haulage or rock removal**

As indicated in section 2.3, the planned rock handling system comprises diesel powered loaders from the stope to truck loading points and diesel truck haulage to surface. The truck haulage from 200 m below surface to surface based on a production of 0.24 Mtpa is approximately 0.38 Mtpa.km and the required truck haulage ventilation rate and value of $\beta$ is about 3 m$^3$/s when using a specific ventilation rate of 7.5 m$^3$/s per Mtpa.km.

Based on a 15 t truck with a 164 kW engine (typical Toro TH 315), for a 20 000 tpm production rate, approximately 60 cycles per 24 h are required. For a round trip of 4 km (some lateral haulage) and an average speed of 10 km/h, each cycle takes 24 minutes. Assuming 20 h per day is available for hauling rock, 50 cycles per day are possible. It is therefore unlikely that more than two trucks will be hauling at any time.

It is therefore not necessary to make an additional air quantity allocation for the decline haulage with dilution ratios about nine times that required for two trucks and a limited number of loaders operating.

**3.3 Summary of ventilation requirements by activity**

The total ventilation requirements based on expected activities is:

- Development: 30 m$^3$/s
- Mining (stoping): 120 m$^3$/s
- Haulage: nil m$^3$/s
- Mine total: 150 m$^3$/s

This is sufficiently close to the 125 m$^3$/s obtained using the indicative values for the 150 m$^3$/s to be used as the basis for the ventilation system design.
4 VENTILATION SYSTEM DESIGN

4.1 Optimum airway sizes

The background to determining economic airway sizes and air velocities is given in Appendix 3. Generally there is a relatively wide range of acceptable economic air velocities where changes in air velocity have only a small effect on the overall costs. The approach taken is to determine an acceptable range of economic velocities where the minimum financial criteria for the project are met as well as the optimum value that maximises the rate of return or net present value.

Where mine openings are used for other purposes as well as ventilation such as access declines, any limiting constraints indicated by the design criteria must also be considered. The following analyses are based on electric power at US$ 0.10/kWh, a fan efficiency of 70%, main installed fan costs at US$ 1500/kW, annual fan maintenance at 15% of the operating power costs, an interest rate of 8%, a life of 7 years and a tax rate of 35%.

Resistance to airflow

Conventionally in mining, the Atkinson equation is used to determine the pressure losses in mine airways and the expression used is:

\[ Pd = \frac{K C L Q^2 \rho}{A^3 1.2} \]

where  
\( K = \) Atkinson friction factor (Ns²/m⁴) 
\( C = \) circumference or perimeter of airway (m) 
\( L = \) length (m) 
\( Q = \) air quantity (m³/s) 
\( \rho = \) air density (kg/m³) 
\( A = \) cross sectional area of airway (m²)

Values for the applicable Atkinson friction factor can be obtained from the available literature and by measurement. An alternative approach to estimate airway resistance is to use the Darcy-Weisbach relationship to calculate the pressure loss (Pd) for a given airflow:

\[ Pd = \lambda \frac{L \rho V^2}{D_h 2} \]

where  
\( \lambda = \) Friction factor (dimensionless) 
\( D_h = \) Hydraulic diameter (4 x area/perimeter, m) 
\( V = \) air velocity (m/s)

The friction coefficient \( \lambda \) can be obtained from the Colebrooke-White relationship:

\[ \frac{1}{\sqrt{\lambda}} = 1.74 - 2 \log \left( 2 \varepsilon + \frac{18.7}{\text{Re} \sqrt{\lambda}} \right) \]

where  
\( \varepsilon = \) relative roughness (absolute roughness/hydraulic diameter, dimensionless) 
\( \text{Re} = \) Reynolds number (dimensionless)

For most underground airways, the Reynolds number is greater than \( 10^6 \) and the second term in the brackets is small enough to be ignored. An exception to this would be airflow through
a caved zone where laminar flow may exist. A relationship between the friction coefficient $\lambda$ and the Atkinson friction coefficient $K$ is obtained by equating the Darcy-Weisbach equation to the Atkinson equation and results in:

$$K = \lambda / 6.667$$

**Declines and lateral airways**

Typical values of the fixed and variable components of the excavation cost (US$/m) of a decline or lateral airway are:

$$900 + 150 A$$

Where $A$ is the cross sectional area ($m^2$)

The results of the analysis described in Appendix 3 using the above costs and for an airflow of 100 m$^3$/s results in an optimum air velocity of approximately 8.0 m/s. The maximum and minimum airway velocities where the minimum financial criteria are still met are 5.0 m/s and 12.5 m/s. The minimum, optimum and maximum air velocities are to a small extent affected by airway size and an airflow of 100 m$^3$/s is used as being representative for the Namib North mine ventilation system design.

For example, for a 3.2 m x 3.5 m decline with a cross sectional area of 11.2 m$^2$, the minimum and maximum air flow rates are approximately 55 m$^3$/s and 140 m$^3$/s and the optimum quantity at the optimum air velocity of 8.5 m/s is 95 m$^3$/s. Using the above relationship, the costs of a 3.2 m x 3.5 m decline is approximately US$ 2 600/m. Using a 6.0 m/s limiting air velocity guideline value to avoid excessive dust pick up, the air carrying capacity is reduced to about 70 m$^3$/s.

**Raise bore holes**

Typical fixed and variable components of the excavation cost per metre (US$/m) of raise bore holes (RBH) are:

$$2500 + 75 D^{2.75}$$ for raise diameters (D) 4.0 m and larger (surface rigs)

$$900 + 250 D^{2.2}$$ for raise diameters (D) 3.6 m and smaller (underground)

Where $D$ is the diameter of the RHB (m)

The results of the economic analysis based on airflows of 100 m$^3$/s and including typical shock losses results in optimum air velocities of 11.5 m/s and 15.5 m/s for large and small raises respectively. The minimum and maximum air velocities were 7.5 m/s and 16.5 m/s for the larger raises and 10.0 m/s and 22.5 m/s for the smaller raises.

It is unlikely that there is any scope for the larger raise bore holes at Namib North mine and 2.4 m diameter raise bore holes would have a cost of about US$ 2650/m

**Long hole raises**

As an alternative to raise bore holes between sub levels, with a spacing of about 30 m it is usually practical to use long hole raises (LHR) between the sub levels. If the raises are all offset, the system has a very high shock loss component. This can be reduced where the ore body is vertical or near vertical by having as many raises as practical collinear which requires excavation from the bottom raise upwards. For simplicity, the raise costs are taken to be the same as for lateral airways.
Shock losses are still high even with collinear raises and equal to as much as 3.5 times that of the raise alone. When taking this into account, the optimum air velocity is 6.0 m/s and the minimum and maximum air velocities are 3.5 m/s and 9.0 m/s. For an airflow of 100 m³/s, the optimum raise size has a cross sectional area of 16.5 m². In a 3.5 m wide access, drilling constraints restricts this dimension to about 3.0 m wide, and because of the limiting sizes of any connecting airways, a practical size is equivalent to about 3.0 m x 5.5 m.

The costs of a 25 m long, 16.5 m² LHR is about US$ 85 000.

4.2 Ventilation system arrangements

With up to 20 mining and development activities, several levels both to the north and south of the ore zones access require ventilation at any one time. Within the mining areas, the decline(s) and a parallel intake shaft supply air to the levels. Depending on the location of the stopes and other activities, the intake air will flow north and south with auxiliary ventilation systems supplying the “dead end” activities such as stope access development. Exhaust from the stopes on the upper levels passes through the upper levels to the worked out stopes (Junction stope) connected to surface as illustrated in Figure 4.1.

At a power cost of US$ 0.10/kWh, the annual cost of power is US$ 876/kW. Using a cumulative present value factor of about 3.5, the indicated life of mine cost of 1 kW of main fan power is approximately US$ 3000.

4.3 Intake airways

Surface to the lower levels
As indicated in Figure 4.1 the North and South declines are scheduled to be used as intake airways between surface and 7 level along with the surface shaft. For this 2.0 m x 3.0 m shaft, the optimum, minimum and maximum air quantities are 50 m³/s, 30 m³/s and 75 m³/s respectively. With one 3.2 m x 3.5 m decline and the shaft, an intake air quantity of 150 m³/s is possible within the optimum range however the decline air velocity is greater than 6.0 m/s and additional dust mitigation measures would be required such as water sprays over the full decline length.
Alternative arrangements of declines, decline size and the shaft are summarised in Table 4.1 and are all based on a free split. For a mine depth of 200 m and an overall main fan efficiency of 70%, the difference in main fan power between option 1 (existing decline and intake shaft) and option 5 (two large declines and intake shaft) is approximately 125 kW. The increased power cost would be about US$ 375 000 and most unlikely to justify the additional main fan and excavation costs.

It is concluded that ventilation requirements and economics do not justify additional airways as declines or raises and these should be motivated for other reasons such as reducing haulage distance and providing escape-ways.

### Table 4.1 – Intake airway arrangements

<table>
<thead>
<tr>
<th>Options</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Intake pressure loss (Pa/100 m)</td>
<td>395</td>
<td>261</td>
<td>191</td>
<td>142</td>
<td>110</td>
</tr>
<tr>
<td>Shaft 2 m x 3 m</td>
<td>84 m³/s</td>
<td>69 m³/s</td>
<td>60 m³/s</td>
<td>51 m³/s</td>
<td>44 m³/s</td>
</tr>
<tr>
<td>S decline 3.2 m x 3.5 m (67 m³/s)</td>
<td>66 m³/s</td>
<td>45 m³/s</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>S decline 3.5 m x 4.5 m (67 m³/s)</td>
<td>81 m³/s</td>
<td>53 m³/s</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>N decline 3.2 m x 3.5 m (95 m³/s)</td>
<td>45 m³/s</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>N decline 3.5 m x 4.5 m (67 m³/s)</td>
<td>60 m³/s</td>
<td>53 m³/s</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The values given in brackets in Table 4.1 are the air quantities based on a limiting air velocity of 6 m/s. There are two issues with respect to the size of a decline, the first is dust pick up where air velocity is normally limited to about 6.0 m/s unless dust mitigation measures are applied. Generally, the application of water sprays is sufficient mitigation for air velocities up to about 9.0 m/s and, for air velocities in excess of this, footwall consolidation would also be necessary.

The second issue is the size of the decline in relation to ventilation during development and the clearance for the trucks. To achieve a haulage rate of 0.25 Mtpa it is most likely that 15 tonne trucks will be required such as the 164 kW Toro TH315. The overall dimensions of the truck are 2.25 m wide and 2.35 m high and these would probably be suitable for haulage in a 3.2 m x 3.5 m decline. Allowing 0.2 m for the road bed, clearances both beneath and above the duct would however limit the development duct size to 0.55 m in a 3.5 m high decline.

If a new decline is to be developed, the minimum decline height would be 3.75 m when using twin 760 mm ducts or about 4.0 m when using a single 1070 mm diameter duct.

### 4.4 Exhaust airways

**Junction stope**

Junction stope is a mined out area from 290 elevation to about 120 elevation and, from the 5 m interval “plan slices” provided as “Junction slices” in the PowerPoint file provided, the cross sectional area varies from about 75 m² to about 200 m². The pressure loss in Junction stope based on a roughness of 500 mm and the minimum area is about 1 Pa/100 m for an exhaust airflow of 150 m³/s and is negligible. There is a small connection from the top of the stope through the crown pillar to surface.

This connection through the crown pillar can be increased in size and, based on an optimum air velocity of about 8 m/s, the indicated cross sectional area is about 18 m². The actual size and configuration will be determined by the main exhaust fan arrangements. Assuming a 20 m length of raise through the pillar and shock loss factor of 1.0, the pressure loss for an airflow...
rate of 150 m$^3$/s is estimated to be about 58 Pa i.e. an overall pressure loss of about 60 Pa.

**Shrinkage stopes to Junction stope**

The pressure loss between the top of the shrinkage stopes and Junction stope will depend on the production plan and the number of exhaust or collection airways. These airways are assumed to be 3.2 m x 3.5 m in size and a maximum of 400 m from the Junction return. With both north and south mining areas the minimum number of exhaust air ways is two and the pressure loss over 400 m for 75 m$^3$/s (150/2) would be 258 Pa.

It is far more likely that with the number of ore shoots (stoping lines) that there would be at least four available returns. Where necessary, some of the upper levels can be connected between levels with relatively short length long hole raises to ensure that there are a sufficient number of return airways to Junction stope. Assuming at least four returns for the mine, the pressure loss would be 65 Pa.

**Pressure losses in a stope line**

A feature of shrinkage stoping is the variable cross sectional area for airflow caused by blasting breasts/benches followed by draw down of the “swell”. The contractions and expansions created cause shock losses leading to higher pressure losses. Assuming that the velocity is limited to a maximum of 4 m/s (determined by the main fan pressure available), the pressure losses in each contraction and expansion is about 10 Pa (velocity pressure $VP = 1.2 \times 4^2/2$ Pa).

It is most unlikely that there would be more than ten contractions/expansions in a stope line and that each would have a maximum 4 m/s velocity. Consequently, the pressure losses in a stope line between the bottom intake and the top exhaust are likely to be between 10 and 50 Pa.
5. FAN SELECTION

5.1 General and pressure losses

Network simulations
Many mine networks obtained from planning exercises (as opposed to the airways actually mined) are large with many hundreds and sometimes thousands of airways particularly when using mine design packages to establish the network. The use of such large networks for routine planning purposes is both cumbersome and time consuming with many opportunities for error. The main use of a very large network is first to estimate the actual network as accurately as possible leading to heat simulations where these are required.

At this stage in the planning process for Namib North mine with the inherent uncertainties in airway arrangements and other details, the use of detailed network simulations infers an unjustified accuracy. In metalliferous mine networks over 90% of the mine pressure loss (and therefore main fan pressures) is in the main intake and return airways. Since these constitute between 5% and 20% of the total number of branches and some of them can be combined into a simple series or parallel arrangement, a simplified or skeleton network can be created instead.

Although it may appear that some accuracy is sacrificed, when the skeleton is thoughtfully constructed, the results produced are well within the normal tolerances involved in ventilation planning. By the nature of creating a skeleton, several “levels” can be combined and the air quantity is allocated to the bottom level i.e. the pressure loss is not underestimated. This, along with the use of a minimum number of leakage flows usually results in acceptable overall pressure estimates.

The planned Namib North mine network is very simple with parallel declines and the shaft as intake between surface and the lowest mining level. Similarly, the exhaust system is straightforward and comprises collection levels (possibly with connecting long hole raises) from the northern and southern sections connected to the Jubilee stope as the main exhaust with a long hole raise through the crown pillar to surface. The air velocity in the shrinkage stopes from the intake decline/level to the collection level would be limited to 4 m/s and the pressure loss would be of the order of 50 Pa for the highest airflow rate. The estimated pressure losses are identified in sections 4.3 and 4.4 and summarised in the following section.

5.2 Main fan arrangements

Main fan duty specification
Based on the pressure losses identified in the previous sections a summary of the fan duties is given in Table 5.1. The base case has the existing (3.2 m x 3.5 m) haulage decline and the shaft as intake and the Jubilee stope as exhaust with a raise through the crown pillar.

<table>
<thead>
<tr>
<th>Airways</th>
<th>Quantity (m³/s)</th>
<th>Pressure losses (Pa)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>One decline</td>
</tr>
<tr>
<td>Intake airways</td>
<td>150</td>
<td>790</td>
</tr>
<tr>
<td>Workings (shrinkage stopes)</td>
<td>150</td>
<td>50</td>
</tr>
<tr>
<td>Collector levels</td>
<td>150</td>
<td>65</td>
</tr>
<tr>
<td>Exhaust to surface</td>
<td>150</td>
<td>60</td>
</tr>
<tr>
<td>Fan pressure (Pa)</td>
<td>150</td>
<td>965</td>
</tr>
<tr>
<td>Fan power (total kW)</td>
<td>150</td>
<td>210</td>
</tr>
</tbody>
</table>
Main fan arrangements

It is assumed that the main fans will be installed on surface above Jubilee stope. With a limited mine life of less than 10 years, a relative low pressure system and reasonable power costs, the main fans do not have to be designed for a long life and a high efficiency. The duty is suitable for axial flow fans and this can be used to reduce overall costs by selecting a much simpler inlet box arrangement.

The actual fan cost, that is the section with the impeller and the motor and drive, is usually about US$ 500/kW of installed fan power. The additional costs are the fan outlet diffuser and the fan inlet arrangements. For a fan handling 75 m³/s (two fans used to ensure adequate redundancy), the fan impeller section is likely to be 1.8 m in diameter. If the diffuser outlet velocity is reduced from 15 m/s to 10 m/s, the theoretical reduction in total fan power is 16 kW. There are losses in the diffuser which would reduce the saving in fan power to about 13 kW which has a life of mine value of about US$ 39 000 (13 x 3000).

A diffuser suitable to reduce the outlet velocity to 15 m/s is approximately 4 m long with an outlet diameter of 2.5 m. The mass of steel required for two diffusers is about 5.0 tonnes with a cost of about US$ 50 000. A diffuser suitable to reduce the outlet velocity to 10 m/s is approximately 7.5 m long with an outlet diameter of 3.1 m. The mass of steel required for two diffusers is about 10.8 tonnes with a cost of about US$ 108 000. The additional US$ 58 000 in diffuser cost (ignoring any foundations and installation cost) is greater than the estimated saving of US$ 39 000 and therefore not justified.

The standard fan inlet arrangement is to have a bend out of the raise followed by a bifurcation and inlet duct into each fan. The mass of steel required is approximately 17 tonnes at an estimated cost of US$ 170 000. Two inlet boxes with back draft dampers would have a mass of about 2.5 tonnes and an estimated cost of US$ 25 000. The pressure loss in an inlet box would be no greater than 0.6 VP which is about the same as for a shaft bend (0.35 VP) with bifurcation and fan inlet duct (0.2 VP). There is no financial advantage to using the more expensive inlet arrangement.

As exhaust axial flow fans the blades could be constructed from standard stainless steel to minimize the effects of corrosion mainly from diesel fumes. Alternatively, with a relatively short mine life, replacement impellers could be considered. The estimated cost for the twin main fan installation is:

- Fans (two 75 kW at 500/kW) US$ 75 000
- MCC and starters 60 000
- Electrical supply 25 000 (assumed)
- Additional steelwork (7.5 tonne) 75 000
- Concrete (10 m³) 10 000
- Base cost US$ 245 000
- Tender and contract (10% of base cost) 24 500
- Install and commission (17.5% of base cost) 42 900
- Total cost US$ 312 400

The tendered cost of the main fan for a Bulgarian mine with an inlet box varied from a low value of US$ 645/kW (Zitron) to US$ 1815/kW (TLT) with an intermediate value of US$ 890/kW (Howden) when adjusted for inflation. On a comparable basis, the Namib North mine fan costs are estimated to be US$ 210 000 or US$ 1400/kW. Budget prices for Namib North mine are being requested and as soon as they are available, this section will be updated.

The main fan operating power costs assuming the two decline ventilation system i.e. 150 kW of
fan power is US$ 0.13 million per year. Fan maintenance costs would be approximately 15% of
the operating power cost or an additional US$ 0.02 million per year.

5.3 Auxiliary fan requirements

Development fans
For long developments (declines and exploration drives), the fan requirements per heading are
either two 30 kW fans when using twin 760 mm ducts or one 45 kW fan with 1070 mm duct.
The auxiliary electric fans costs including starters but excluding the power supply are
approximately US$ 350/kW or US$ 10 000 each for the 30 kW fans and US$ 16 000 for each of
the 45 kW fans.

For the short developments where the maximum length from through ventilation is 250 m, the
fans required using 760 mm duct are 15 kW and have an estimated cost of US$ 5000 each.
Alternatively, compressed air fans could be used such as the Korfmann DV6 at a delivered cost
of about US$ 8000. The fan has a compressed air consumption of 0.07 m³/s or about 40% more
than a standard rock drill used with a jack leg.

Although a separate electrical supply is not required when using compressed air fans, the fan
efficiency is low at between 10% and 12% compared to 65% to 70% for electric powered
auxiliary fans. The overall system efficiency could be as low as 3% to 4% when compressed air
leakage and pressure losses in the distribution system are taken into account.

Overall auxiliary fan requirement
The overall number of auxiliary fans necessary and the underground power requirements
depend on the development schedules. As indicated in section 3.1, the overall costs are probably
between 8.0 and 14.0 US$/m per m³/s delivered to the face. Taking average values, the
ventilation costs of long development are approximately US$ 110/m and short development
US$ 75/m. The development ventilation costs include duct, fans and power.

5.4 Effect of increased mine depth

Increasing mine depth mainly affects the intake system and possibly the return system from the
lower levels into Junction stope. With reference to Tables 4.1 and 5.1, the increase in intake
pressure loss based on two 3.2 m x 3.5 m declines and long hole raises below the shaft is
approximately 190 Pa for a 100 m extension. In respect of the exhaust to Junction stope, there
are likely to be sufficient additional routes for the exhaust air as stopes are completed.

It is possible that the mine pressure loss at the increased depth would increase by about 200 Pa
requiring an additional 45 kW of main fan power. With an appropriate initial fan selection, the
increased duty can be achieved by replacing the 75 kW drive motors with 110 kW motors and
changing the fan blade settings.
APPENDIX A1 - DESIGN CRITERIA

The amounts of ventilation and cooling required for mines where heat may be a problem underground is normally determined by offsetting the higher capital and operating costs associated with increasing ventilation and cooling against the benefits and reduced operating costs from improved safety and productivity. An optimum design is considered satisfactory provided that the resultant underground environmental conditions meet defined minimum constraints or mine occupational health standards. These may be determined from either the legal framework in which the operations take place such as legislation or codes of practice or may simply derive from good and acceptable practice.

Where heat is not expected to be an underground problem such as the Namib North Mine, the design is much simpler with ventilation being used to meet the minimum constraints and the optimisation is limited to ensuring that the most economic airway and fan arrangements are adopted. Typical design criteria that may be considered are:

- Minimum ventilation air quantities and velocities
- Dust concentrations and exposure
- Noxious gas concentrations and exposure
- Diesel powered equipment exhaust dilution
- Explosive gas/dust concentrations
- Heat stress control criteria
- Ionising radiations exposure
- Control of noise for hearing loss and speech interference

When reviewing the ventilation design for a new mine or an extension to the current mine workings of an existing mine, it is necessary to consider the applicable design criteria and, by analysis and simulations where appropriate, it can then be demonstrated that the minimum constraints/mine standards will be met. An attempt should also be made to assess whether there may be future changes in the minimum design criteria that may subsequently impact on the ventilation and/or cooling standards and design. These should be examined critically and placed in context with any recent changes.

Some absolute design criteria will not change - an explosive gas mixture is an explosive gas mixture; however, a minimum dilution rate might vary in order to provide a different (normally greater) margin of "safety". With other respirable pathogens there has been a consistent reduction in threshold limit values with time, which is partly a result of the changing acceptability of different levels of potential biological damage and partly the increased knowledge resulting from epidemiological studies.

More recent recommended changes in threshold limit values (respirable quartz, diesel particulate matter and nitrous oxides) are not based on sound epidemiological data and instead on an overemphasis of the indicators from other techniques (in vivo/in vitro) resulting in threshold limit values based on what is often known as “junk science”.

For the Namib North Mine, the design criteria for explosive gas/dust concentrations, heat and heat stress and ionising radiations exposure should not be applicable and are not considered in detail in this appendix.
A1.1 Air Quantity and Velocity

Background
A general duty of care is that ventilation should be provided and maintained to remove or dilute airborne contaminants to an acceptable level where there is no known hazard to health. Within current statutory requirements there is usually no general minimum air quantity required and the only specific air quantity values are given for diesel engine exhaust dilution. In the past, minimum air quantities were specified and mainly concerned with ensuring that there was sufficient oxygen in the air supplied to working places. Generally, it is required that carbon dioxide concentrations should be less than 5000 ppm and the oxygen content should be greater than 19% (SA Occupational Exposure Limits).

There is also a case for a minimum air velocity particularly where heat stress may be a problem. The values given in Table A1.1 are the clothing corrected air-cooling power at varying wet bulb temperatures and air velocities (Howes, 2014a). The radiant temperature is 2°C higher than the dry bulb temperature which is 8°C higher than the wet bulb (typical of conditions where diesel powered equipment is used). The barometric pressure is 100 kPa and the clothing regime is typical for work at 0.52 clo (short sleeved shirt, trousers and protective head and foot ware).

An air velocity of 0.1 m/s is the upper limit for natural convection over the human body and reflects what may be expected with no discernible airflow. An air velocity of 0.25 m/s is the minimum allowed for mining activities in many countries such as South Africa and 0.5 m/s is the minimum permitted where the wet bulb temperature exceeds 25°C in Western Australia. Typical minimum air cooling powers to achieve thermal equilibrium at different levels of physical activity are; rest 50 W/m², light work (equipment operator) 110 to 120 W/m², medium work (light construction and jack leg drilling) 140 to 170 W/m², hard work (heavy construction) 200 to 250 W/m² and very hard work (shoveling) 250 to 350 W/m².

### Table A1.1 - Air cooling power (W/m²)

<table>
<thead>
<tr>
<th>Air velocity (m/s)</th>
<th>Wet bulb temperature (°C)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>20.0</td>
</tr>
<tr>
<td>0.1</td>
<td>207</td>
</tr>
<tr>
<td>0.25</td>
<td>253</td>
</tr>
<tr>
<td>0.5</td>
<td>291</td>
</tr>
<tr>
<td>0.75</td>
<td>310</td>
</tr>
<tr>
<td>1.0</td>
<td>322</td>
</tr>
</tbody>
</table>

- Values < 115: Stop work
- 116 to 140: Re-deploy
- 141 to 175: Cautionary
- > 176: No restriction

Recommendation
The mine is located close to Rosh Pinah where the climate is extremely arid (see section 2.2). During the warmest months, the 2.5% surface wet bulb temperature at Namib North is unlikely to exceed 21°C and underground wet bulb temperatures of between 24°C and 26°C could be expected in a shallow mine (about 200 m deep) where diesel powered equipment operates. It is evident from the values given in Table A1.1 that an air velocity of 0.50 m/s is necessary for a medium work rate to be safely carried out.
A1.2 Dust and dust sampling

Summary

The purpose of this section is to provide background to the development of dust exposure assessment and to support recommendations in regard to sampling and record keeping.

Essentially, prior to the late nineteenth century the dust encountered in mining was no worse than other industrial occupations and although silicosis was known it was not a major hazard to health. The introduction of pneumatic drilling (replacing hand drilling) in the late nineteenth century resulted in very high dust levels and significant increases in silicosis. The problem was tackled by controlling the dust (mainly wet drilling) and dilution ventilation and by sampling to determine dust exposure levels.

It was not until the middle of the twentieth century that the causes of silicosis were sufficiently understood to determine the best method of exposure assessment. The early sampling methods were quite crude and did not adequately reflect the hazard to health. By the late 1950’s research (*in vivo* – white rats) had shown that it was the surface area of the quartz that probably best reflected the silicosis hazard however a suitable sampling and assessment method was not then readily available.

Gravimetric methods developed for coal mines in the 1960’s were adapted for use in non-coal mines in the 1970’s and used to assess statutory respirable quartz exposure from about 1980 onward. Although some research was undertaken to support the change in assessment method, it is not definitive. However, gravimetric sampling is straightforward to undertake and any potential bias or error is small enough not to be significant.

The determination of risk of an occupational hazard requires suitable epidemiological data and, for quartz this has been confounded by two factors; sampling methods and cost. Early sampling methods did not reflect the hazard (mainly number counts with very short sampling periods) resulting in a change to gravimetric in the 1970’s and 1980’s. The cost factor is where several major studies were stopped because the cost of continuing the studies was higher than the cost of compensation for the lower rates of silicosis.

Reliance has therefore been placed on some dubious conversions of number counts to gravimetric values. The ACGIH original derived standard of 0.1 mg/m$^3$ for respirable quartz is almost certainly flawed and standards of less than 0.2 mg/m$^3$ have no demonstrable epidemiological basis that justifies the lower values.

Background and early dust sampling methods

The main difficulty in determining rational gravimetric respirable dust standards is the absence of a definitive epidemiologically derived relationship such as that for ionising radiations. This was a result of uncertainty in the aetiology of silicosis prior to the 1950’s and deficiencies in the early sampling and assessment methods leading to doubt in regard to the actual exposures of personnel involved in any studies.

The recommendations from the 1959 Johannesburg Pneumoconiosis Conference (Orenstein, 1960) in regard to quartz dust was to sample the respirable fraction, to sample for longer periods (the working shift) and to measure the surface area of the respirable fraction. For coal dust the mass of respirable dust was to be measured.

The sampling method initially used in South African mines was the sugar tube (gravimetric method, total dust sampled) where sugar was used to filter out dust from a 10 minute sample. The konimeter (impingement method, short duration sample, < 5 μm by microscopy) was introduced in 1916 although treatment of the slides for assessment was only standardised in
Thermal precipitator (thermophoresis, microscopy) was introduced in 1935 and became a standard for comparison of other methods. The TP sampling time was usually three hours and a long running version used for 8 to 24 hours. A modified thermal precipitator (MTP) was developed in 1950 with a 10 minute sample time and multiple samples possible on a standard microscope slide. This was to be combined with the diffraction size-frequency analyser and used to routinely obtain measurements of the surface area of the respirable dust. The Greenburg-Smith impinger (impingement method, microscopy assessment) was developed in 1922 in the US and dust is caught by impinging air at high velocity onto a small plate immersed in water. It was modified to the midget impinger in 1928 after the Vermont granite sheds sampling. The assessment also changed to light-field microscopy with a micro-projector where particle sizes of between 1 µm and 5 µm are counted and isopropyl alcohol was used instead of water to collect the dust. Personal gravimetric respirable dust sampling methods were developed for coal mines in the 1960’s and investigations started in the early 1970’s for their use in non-coal mines. Sampling and quartz assessment methods were sufficiently developed by the late 1970’s for gravimetric respirable dust and quartz methods to be adopted for non-coal mines. 

Respirable quartz content
Differential fracture characteristics between quartz and other minerals as well as weaker grain boundaries normally result in the respirable fraction of the dust being less than and about one third that present in the host rocks. Silica combined with other elements to form silicate minerals such as feldspar does not have the same “active” lung effect as free crystalline silica. For example, granites may contain up to 65% silica however, depending on the type of granite, between 25% and 35% of this is free crystalline silica or quartz. The amount of quartz present in respirable dust from a granite source would then usually be about 10%. This approximate relationship has been found from sampling in many mines including Cornish tin mines which are in granite similar to Vermont. With the exception of some early sampling where the assessment of respirable quartz was still being developed, the approximately one third value is typical of nearly all mine sampling results in the available literature with one exception that relates to coal mines (Howes, 2014b).

There are two other factors that may also affect the proportion of quartz in the respirable dust relative to the host rock that depends on the sampling and assessment method. These are lubricating oil and diesel particulate. Lubricating oil is invariably present as an aerosol in significant amounts where pneumatic equipment is used extensively such as for rock drilling. The proportion of lubricating oil in respirable dust can vary between 0 and 90%. The amount of diesel particulate in respirable dust will depend on many factors and principally the diesel utilization and diesel exhaust dilution rates. Typical values would vary between 0.1 and 1.0 mg/m³. The amount of oil mist and diesel exhaust particulate is usually determined by introducing an additional weighing procedure after heating the filters and driving off volatiles. This technique is not usually compatible with subsequent X-ray diffraction or infrared measurements for actual quartz content (see later).

The effect of lubricating oils and diesel particulate is of limited relevance when the actual respirable quartz is directly measured, the importance is when retrospectively attributing respirable quartz exposure values to dust samples taken earlier. Respirable dust and quartz concentrations obtained prior to 1970-75 when methods and techniques were being
developed should be treated with caution. There is also a problem of accuracy of assessment particularly for low respirable dust concentrations of less than 0.5 mg/m³.

**Mass and surface area**

Silicosis research with animal studies was first used in the 1930’s. In order to develop the research along quantitative lines, injection methods were used to introduce a known dosage into the lung tissue. However, it was recognised that these methods are removed from the industrial conditions which result in human exposure. There are complications arising out of maintaining a dust cloud, possibly different breathing rates of animals and lung retention characteristics which are all avoided using injection methods.

From the initial trial undertaken in the mid 1950’s it was concluded for equal mass, the fibrogenic activity increased considerably with decreasing size. The optimum fibrogenic size of quartz in this trial was found to be between 1 μm and 2 μm equivalent diameter. Difficulties in interpretation of lung sections to provide silicosis grading were recognised and additional work undertaken. The second trial investigated whether the increased fibrogenicity of the 1-2 μm particles could be demonstrated (Howes, 2009).

Two quartz particle size fractions (0.2-2 μm and 2-5 μm) of equal surface area (600 cm²) were intratracheally injected into a statistically significant number of rats. The rats were killed after 80 days and the lung sections examined and graded as in the first trial. The results were inconclusive, with similar degrees of fibrosis arising from both size fractions. The author felt that there was some evidence (based on number of lesions) that the larger size fraction may have been more fibrogenic.

The third trial undertaken in the mid 1960’s investigated the effects of three size fractions by intratracheally injecting equal surface areas (600 cm²) of quartz into rats and evaluating the lung sections after 120 days. The preparation of the size fractions, determination of size distributions, dosing procedure and assessment of results were similar to the second trial. The calculated surface areas were actually 530, 387 and 535 cm² for the < 1 μm, 1-3 μm and 2-5 μm fractions respectively.

The conclusion from the last trial is used as justification for gravimetric sampling where respirable quartz is the main hazard to health. Of concern in these trials is inconsistency in measurement of particle size, an absence of fine particles in the larger size fractions and discrepancies in the calculated mass and surface area injected. In addition there is no accounting in the trials for pulmonary retention, as not all the injected quartz would remain within the lungs.

In all three trials, the finest size fraction was measured using an electron microscope and the others using light microscopy. It is accepted that the minimum particle size that can be classified optically is approximately 0.5 μm. In the case of the electron microscope, the resolving power is two orders of magnitude higher and, it is therefore probable that, in the listed size distributions for the coarse fractions, the fine particles that may be present have been ignored. The implication is that the surface area of the finest fraction relative to the coarser fractions was overestimated and a greater mass should have been injected.

This latter point is true if there were fine particles in the coarse fractions. The method of separating the different size fractions was by sedimentation and centrifuging. When a particle is sufficiently small, collisions with individual fluid molecules displace the particle by a measurable amount. This random or Brownian motion is equal to the settling velocity of a 1 μm sphere (ρ = 2650 kg/m³) in distilled water. Although the settling velocity of particles is increased during centrifuging, the effect of Brownian motion will always result in small particles being present in the coarser size fractions.
The conclusions from the first trial were specific. The degree of silicosis was related to particle size and/or surface area and the high fibrogenicity of the 1-2 μm fraction. These were the basis for the 1959 Pneumoconiosis Conference recommendation of measuring surface area for respirable quartz. From the subsequent trials, it was concluded that the degree of silicosis was probably not related to the surface area.

The mixture of size fractions, masses injected, duration and grading of silicosis obscures a comparison between the trials. A simplistic and qualitative assessment was conducted based on the data from the three trials (Howes, 2009). The aerodynamic mass median diameter (AMMD) was calculated for each size fraction and also a weighted fibrosis grading was estimated. It was concluded that there is no strong evidence for using surface area or gravimetric exclusively and other sampling techniques such as light-scattering, could also give a suitable indication of the hazard to health in regard to silicosis.

Background to respirable quartz standards and number to mass conversions

As indicated above, routine gravimetric sampling was introduced into most non-coal mines by the early 1980’s (Canada in 1981 although introduction in South Africa was delayed until 1992) and has provided less than 30 years of measured exposure. A definitive epidemiological relationship between silicosis and the mass of respirable quartz obtained gravimetrically is therefore unlikely however it is possible that some of the previous number count data could be converted into equivalent gravimetric values.

Dust in mining is a result of attrition processes such as grinding and crushing of rock. These are described by a log-normal size distribution and confirmed by measurements taken in underground mines. A feature of log-normal distributions is that the number, surface area and mass distributions are all log-normal with the same geometric standard deviation (σG). Conversion from one distribution to another is straightforward.

Attempts have been made to determine number to mass concentration conversions. In the UK for the 25-pit epidemiological study, the MNI (Mass Number Index – mass in mg/m divided by number count in 1000 ppc using the TP) was found to vary from 10 to 30. This was considered to be too large and parallel number and mass sampling continued. In addition to the size distribution, the variation in MNI for the UK coal mining industry was also found to depend on the instrument used and the method used to sample the respirable fraction (cyclone, horizontal elutriator etc).

The UK coal mine dust regulations of 1975 introduced a standard of 0.45 mg/m³ of respirable quartz which included the continuance of personnel X-ray surveillance. There is no standard in the UK for respirable quartz in non-coal mines although a 0.3 mg/m³ limit of respirable quartz was adopted in the British Standards Institute (BS 6164) Code of Practice for tunnelling in 1982 and reaffirmed in 2001. This has subsequently been reduced to the COSHH value of 0.1 mg/m³ in the 2011 revision.

In South Africa, the main epidemiological study started in the early 1960’s was not completed (abandoned after about 10 years) and also used the thermal precipitator (TP and MTP) as the main sampling method. Surface area was measured using a diffraction size-frequency analyser and also respirable surface area and mass were calculated from the TP size distributions. Parallel sampling with the konimeter was also undertaken.

The konimeter, in addition to a short sampling period, was known to have a wandering bias and to underestimate high dust concentrations. Parallel sampling confirmed this with respirable masses (calculated) relative to konimeter counts of 0.2 mg/m³, 0.45 mg/m³, 0.75 mg/m³ and 1.05 mg/m³ for 100 ppc, 150 ppc, 200 ppc and 250 ppc (particles per cubic
The respirable quartz mass would have been about 20% to 25% of these values based on subsequent gravimetric sampling and assessment.

In South Africa a respirable quartz standard of 0.1 mg/m$^3$ of was introduced in 1992 (when the konimeter was replaced) and most probably based on the ACGIH value existing at that time rather than any of the epidemiological data that had been collected twenty to thirty five years previously.

**The US quartz standard**

In the US, the original TLV of 10 mppcf (millions of particles per cubic foot) was based on impinger dust sampling data and an epidemiological study undertaken in the Vermont granite sheds during the 1920’s. The outcome of this study was that when personnel worked in environments where the dust exposure was less than 10 mppcf (obtained using the impinger), silicosis was eliminated.

Conversion studies were performed in the same granite sheds using the midget impinger and size selective samplers with gravimetric methods. However, these studies took place four decades later in the 1960’s and a question arises regarding the validity of comparing the data obtained earlier, with current gravimetric methods.

There are three sources of error in deriving the US quartz standard involving; the appropriate respirable fraction, the mass to number conversion and the respirable quartz fraction. The first arises from the different respirable fraction definitions and for a range of typical underground dust size distributions, the error can be as large as 12%. For the second, a relationship was determined between the BMRC gravimetric definition of respirable dust and the midget impinger dust count of 5.6 mppcf = 1 mg/m$^3$ with a stated and unrealistic correlation coefficient of 0.99 for 23 measured sets of data.

A full analysis using the appropriate respirable dust definition and a range of typical mine dust size distributions indicates that a single value does not have a universal application. With factors varying between 4 and 9 mppcf per mg/m$^3$ for a range of dust densities and typical size distributions, the 5.6 mppcf per mg/m$^3$ is probably a reasonable average.

Dust from the Vermont granite sheds was originally found to have a quartz content of 35.2% and a size distribution converted from number counts with an AMMD of 3.5 and $\sigma_0$ of 1.5. For the ACGIH respirable quartz limit of 0.1 mg/m$^3$, the quartz fraction needs to be 5% obtained using a conversion factor of 5 ($\rho = 2650$ kg/m$^3$) mppcf per mg/m$^3$ and results in a respirable dust limit of 2 mg/m$^3$ (10/5). A 0.2 mg/m$^3$ limit and a 10% quartz fraction would have been more appropriate.

From published mortality studies on the incidence of silicosis in Cornish tin mines and in metalliferous mining in Western Australia, the application of a 0.2 mg/m$^3$ respirable quartz standard has been demonstrated to eliminate silicosis as a compensatable disease. There is no known epidemiological basis for a standard less than this value including the original ACGIH 0.1 mg/m$^3$ derived value used in the US and further reductions in the standard to 0.05 mg/m$^3$ or even half this value are without justification and vexatious.

**Cancer**

In 1991, respirable crystalline silica (quartz) was listed as; *reasonably anticipated to be a human carcinogen*. This was upgraded in 2000 as; *known to be a human carcinogen*. Subsequent studies have been unable to justify this upgrading at the occupational exposures normally encountered and one of the largest studies (Bochman et al, 2005) concluded:

“Our results show that the observed excess risk of lung cancer among the cohort of silica
exposed workers is more likely due to exposure to other occupational carcinogens such as arsenic and PAH’s rather than due to exposure to respirable silica (presumably means respirable quartz). Therefore, it seems questionable whether effective prevention of lung cancer can be achieved by a reduction of respirable silica only.”

And in a later publication, Bochman et al (2007) further concluded:

“In this case-control analysis of the largest industrial silica cohort world wide, relevant occupational and non-occupational confounders are for the first time systematically and quantitatively considered on the evaluation of the association between crystalline silica exposure and lung cancer. The analysis of this study does not provide any evidence that exposure to silica causes lung cancer in the absence of confounding factors.”

Essentially this work has demonstrated that the adoption of even lower limits for respirable quartz using a possible link between respirable quartz and lung cancer is not substantiated by the epidemiological data. For lung cancer to be a problem when exposed to respirable quartz, silicosis must be present and therefore if the respirable quartz exposure is below that for silicosis to occur, lung cancer from respirable quartz is not a problem.

**Respirable dust sampling and quartz assessment**

For projects that have relatively long life such as mines, it is usually cost effective to undertake the dust sampling and assessment “in house”. In regard to the number of samples, where the silica content of the host rock is greater than 20% probably one full working shift sample per employee per year would be required of which one third of the samples could have a respirable quartz assessment (normally undertaken externally).

The number of samples required is really dependent on the number of different activities and the level and variability of the results from each activity. Generally a minimum of 15 samples per year is required for each activity however this could double if there is a large variation in dust concentrations for the activities with the higher respirable dust concentrations.

Respirable quartz is usually assessed using an infra-red technique with PVC membrane filters. The same filters and samples are used for the respirable dust assessment. Based on the results of parallel sampling it is very doubtful that many laboratories using infra-red spectrophotometry (IRS) can provided unbiased and accurate assessments of the respirable quartz content at low dust and quartz concentrations and it is recommended that a laboratory using X-ray diffraction should be investigated (Howes, 2014).

Because of the inaccuracies and imprecision at low respirable quartz concentrations, it is probably justified to use the respirable dust concentration and a fixed percentage of quartz in the respirable dust based on that of the host rock for most of the samples. Typically the proportion of quartz in the respirable dust is about one third of that in the host rock.

Where the quartz content of the host rock is 2% or less, there is no need to assess the quartz content and a respirable dust limit of 5 mg/m³ is suitable. This is obtained from 0.1/0.02 where 0.1 is the limit for respirable quartz and 0.02 is the 2% limiting quartz content of the host rock. This value would most likely be applied at Namib North mine and preliminary sampling should be undertaken to confirm this.
A1.3 Noxious gas standards

Background
It is presumed that there will be no possibility of ingress of flammable gas and a standard is not required. The main sources of other gases result from the operation of diesel powered equipment and blasting in development and shrinkage stoping. The standards used in South Africa are generally based on the American Conference of Governmental Industrial Hygienists (ACGIH) and are given in Schedules 22.9(2)(a) and (b) of the 2001 Mine Health and Safety Inspectorate Regulations. Three values are used; time weighted average (TLV-TWA), short-term excursion limit (TLV-STEL) and, for some respirable pathogens, a ceiling value (TLV-C).

The two main components of nitrous fumes (oxides of nitrogen) are nitric oxide (NO) and nitrogen dioxide (NO₂). The proportion of nitrogen dioxide in the nitrous fumes is variable and generally about 10% in diesel exhaust gases and 25% in explosive blast gases. Sulphur dioxide results from the combustion of diesel fuel containing sulphur and the contribution is minimised by only allowing fuels with less than 0.25% sulphur to be used (see next section).

Carbon monoxide and carbon dioxide also result from the operation of diesel equipment and from blasting. A summary of the current SA exposure standards for the most commonly encountered gases underground is given in Table A1.2. In the European Community, there are significant changes in the limits proposed for nitrous fumes (NO and NO₂) that may impact on future standards – see section A1.6.

<table>
<thead>
<tr>
<th>Gas</th>
<th>SA-MHSC values</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>TLV-TWA</td>
</tr>
<tr>
<td>Carbon dioxide (CO₂)</td>
<td>5000</td>
</tr>
<tr>
<td>Carbon monoxide (CO)</td>
<td>50</td>
</tr>
<tr>
<td>Nitrogen dioxide (NO₂)</td>
<td>3</td>
</tr>
<tr>
<td>Nitric oxide (NO)</td>
<td>25</td>
</tr>
<tr>
<td>Sulphur dioxide (SO₂)</td>
<td>2</td>
</tr>
</tbody>
</table>

Recommendation
The application of gas measurements normally reflects the method of sampling i.e. short duration rather than continuous where the TWA value is used and a STEL limit must not be exceeded. Methods of continuously measuring gas concentrations are becoming more reliable and cost effective and it will be easier to conform to the time weighted average threshold limit values. This has advantages over short duration sampling by being less disruptive and fully meeting any current or future statutory requirements.

Currently, if a short duration sample exceeds the TWA limit and is less than the STEL limit, there is limited background information to determine whether the TWA will be met and, to be safe, work is usually suspended until the concentration is less than the TWA limit. Continuous monitoring of gas concentrations should reduce the number of times that work is suspended when the TWA is exceeded particularly where a continuous average is available.

The effect of extended shift lengths on standards and whether the TLV’s should be reduced for longer shifts is dealt with in section A1.7.
A1.4 Ventilation standards for diesel equipment

Legislative background

Adherence to the noxious gas limits given in the previous section should ensure safe diesel powered equipment operation underground. The minimum exhaust dilution air quantity is sometimes specified in regulations and varies between 0.03 m/s per kW (Western Australia) and 0.06 m/s per kW (Ontario, Canada also NSW Guideline value) and this must reduce the gas concentrations to below the specified threshold limit values given in section A1.3.

There are usually some provisions for reducing the air dilution requirement; if the diesel engine has special design features and/or exhaust gas monitoring and/or particular operating and maintenance practices and/or the use of low emission fuel or, for allowing other diesel units of small engine capacity and operated intermittently to be disregarded when computing the overall dilution air quantity.

There had been a trend to use an air quality index when assessing the potential effects of combinations of respirable pathogens. This is particularly the case with diesel exhaust gases where, in addition to carbon monoxide and dioxide, oxides of nitrogen and sulphur dioxide, unburnt diesel soot (usually known as respirable combustible dust or RCD) is thought to be of greatest concern. The original work for the air quality index (AQI) was undertaken in 1978 and is the source of the relationship:

\[
\text{AQI} = \frac{\text{CO}}{50} + \frac{\text{NO}}{25} + \frac{\text{RCD}}{2} + 1.5\left[\frac{\text{SO}_2}{2} + \frac{\text{RCD}}{2}\right] + 1.2\left[\frac{\text{NO}_2}{3} + \frac{\text{RCD}}{2}\right]
\]

A value of 3.0 was considered to be the maximum desired level of the index for the working mine atmosphere. The 1.5 and 1.2 were synergistic factors and, along with including the dust terms with the gas terms, accepted in the 1984 review of the original work not to be scientifically supportable. It was then suggested that two equations should be used where the AQI(gas) should not exceed 1.0 and the AQI(particulate) should not exceed 2.0.

\[
\text{AQI(gas)} = \frac{\text{CO}}{50} + \frac{\text{NO}}{25} + \frac{\text{NO}_2}{3}
\]

\[
\text{AQI(part)} = \frac{\text{RCD}}{2} + \left[\frac{\text{SO}_2}{2} + \frac{\text{RCD}}{2}\right] + \left[\frac{\text{NO}_2}{3} + \frac{\text{RCD}}{2}\right]
\]

These have a 2.0 mg/m³ limit for respirable combustible dust (RCD).

Expected exhaust gas concentrations

In most mining areas the dilution requirements for diesel engines has not changed significantly over the last thirty years despite changes in gas threshold limit values. For example, the time weighted average TLV for carbon monoxide decreasing from 100 ppm to 50 ppm in the mid 1970’s and then to 25 ppm in the late 1990’s (although still 50 ppm in South Africa and the Australian limit is 30 ppm). This is mainly because there has been an advance in the technology of both diesel engines and diesel fuel, which has resulted in lower emissions for the same engine power. These advances in diesel engine and fuel technology are expected to continue.

The exhaust emissions from diesel-powered equipment are controlled by a combination of air dilution and exhaust gas conditioning. Assuming that perfect gas conditioning could be achieved, the dilution requirements for carbon dioxide would set the minimum dilution rate at approximately 0.025 m/s per kW (allowable limit 5000 ppm by volume). Assuming no gas conditioning at all, the minimum air dilution rate depends on the gas emissions that are primarily a function of the load cycle and the engine combustion characteristics.
Diesel exhaust gas dilution standards are legislative values or contained in accepted codes of practice. They vary considerably and care must be taken in the interpretation of these standards. Dilution rates are typically 0.04 m/s/kW and these apply at the point of use. The value usually applies to all items of equipment at the rated power output and any leakage must be taken into account. Greater dilution rates of 0.06 to 0.12 m/s per kW are sometimes used however these almost invariably apply to all the workings and allow for equipment diversity and the imperfect distribution of ventilation and include leakage.

**Diesel exhaust gases.**

The emissions that may be expected from the combustion of 1.0 kg of fuel in a diesel engine are summarised in Table A1.3. The basis for this is a fuel analysis of 85% carbon, 14.75% hydrogen and 0.25% sulphur and the gas volumes are all given at standard temperatures and pressure. The first three gases are products of complete combustion and the amount produced is a direct function of the amount of fuel burnt. The remaining gases are either products of incomplete combustion or a combination of the excess oxygen with nitrogen in the combustion air.

The values given Table A1.3 are based on a fuel/air ratio of 0.05 kg/kg and limits in the undiluted exhaust of 1500 ppm for carbon monoxide and 1000 ppm for the oxides of nitrogen (typical values). The diesel exhaust emissions will normally be less and depend on the engine load and speed.

**Table A1.3 - Diesel exhaust gases per kg of fuel**

<table>
<thead>
<tr>
<th>Gas</th>
<th>Mass (kg)</th>
<th>Volume (m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Carbon dioxide (CO₂)</td>
<td>3.12</td>
<td>1.70</td>
</tr>
<tr>
<td>Water vapour (H₂O)</td>
<td>1.33</td>
<td>1.77</td>
</tr>
<tr>
<td>Sulphur dioxide (SO₂)</td>
<td>0.005</td>
<td>0.001875</td>
</tr>
<tr>
<td>Carbon monoxide (CO)</td>
<td>0.0306</td>
<td>0.0263</td>
</tr>
<tr>
<td>Oxides of nitrogen (NO₅)</td>
<td>0.0230</td>
<td>0.0175</td>
</tr>
<tr>
<td>Nitrogen dioxide (NO₂)</td>
<td>0.00335</td>
<td>0.00175</td>
</tr>
</tbody>
</table>

Diesel engines have a thermal efficiency of 33% and, the fuel required to produce 1 kWh is 0.24 kg/h. The minimum exhaust air dilution rates are summarised in Table A1.4.

**Table A1.4 – Minimum diesel exhaust gas dilution rates**

<table>
<thead>
<tr>
<th>Gas</th>
<th>Gas production rate (m³/kg of fuel)</th>
<th>Limit ppm</th>
<th>Dilution rate (m³/s per kW)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Carbon dioxide (CO₂)</td>
<td>1.70</td>
<td>5000</td>
<td>0.0227 (0.0241)</td>
</tr>
<tr>
<td>Sulphur dioxide (SO₂)</td>
<td>0.00188</td>
<td>2</td>
<td>0.0625</td>
</tr>
<tr>
<td>Carbon monoxide (CO)</td>
<td>0.0263</td>
<td>25</td>
<td>0.0583</td>
</tr>
<tr>
<td>Nitric oxide (NO)</td>
<td>0.01575</td>
<td>25</td>
<td>0.0420</td>
</tr>
<tr>
<td>Nitrogen dioxide (NO₂)</td>
<td>0.00175</td>
<td>3</td>
<td>0.0390</td>
</tr>
</tbody>
</table>

Note 1. Corrected for CO₂ present in normal air (300 ppm)

Most underground operations are generally required to use low sulphur (less than 0.25%) fuels and it is evident from Table A1.4 that this, and a threshold limit value of 2 ppm for sulphur dioxide, would normally determine the minimum dilution rate.
The diesel engines used in underground equipment are operated at full load for relatively short periods of time such when fully loaded and accelerating. The values given in Table A1.4 are based on instantaneous assessments and normally threshold limit values allow for time weighted averages over a longer period where higher exposures are permitted providing that the shift average meets both the limit and that excursions above average are acceptable. A suitable system would require "on board" continuous gas monitoring demonstrating that the exposure averages were being met.

From a 1998 review of the exhaust gas sampling results of the loaders on an Australian mine, the average carbon monoxide concentration in the engine exhaust gases was 516 ppm with high and low values of 927 ppm and 296 ppm (the allowable concentration in the undiluted exhaust is 1500 ppm).

For an exhaust gas dilution rate of 0.04 m$^3$/s/kW, the carbon monoxide levels would then range from 9 ppm to 27 ppm with an average of 15 ppm. When taking the average engine load of 65 to 70% over a typical load-haul-dump cycle, the expected range of carbon monoxide concentrations would be reduced to between 6 ppm and 18 ppm. These are typical of the values measured in the working areas of mines where a 0.04 m$^3$/s/kW exhaust gas dilution rate is used as a design value.

A similar trend usually occurs for nitrous fumes including nitrogen dioxide.

**Recommendation**

Depending on the amount of sulphur in the diesel fuel, the current exposure limits for an eight hour working shift when applied to the exhaust gases from diesel engines should be acceptable. It should be emphasised that these conclusions are based on the current use of short duration sampling and the application of the time weighted average limits (TWA) to determine whether the operations can continue or must stop. Where continuous monitoring is possible, the purpose of the time weighted average (TWA) and short term excursion limits (STEL) can be met without disrupting excavation activities.

The recommended minimum gas dilution rate is 0.05 m /s/kW at the point of use and the effects of leakage must be taken into account.

**Diesel particulate**

Prior to the mid 1960’s, the main application for diesel powered equipment in underground excavations was as locomotives for track haulage where the power per unit was usually much less than 100 kW. With the introduction of front end loaders and trucks in the mid to late 1960’s with individual rated powers of between 150 kW and 500 kW, there were concerns with respect to the health effects of particulate in the diesel exhaust gases.

Diesel particulate comprises approximately two-thirds inorganic or non-volatile carbon and one-third hydrocarbons (sometimes known as cyclic hydrocarbons or poly-nuclear aromatic hydrocarbons – PAH or PAC) with trace amounts of metallic compounds. The particles are about 0.1 μm in size however as a result of diffusion, they rapidly agglomerate and in the diesel exhaust, and the resulting particulate has a mass medium diameter $D_m$ of about 0.3 μm and a standard geometric deviation $\sigma_g$ of 6.0.

A 2.0 mg/m$^3$ limit for respirable combustible dust (RCD) was adopted in Canada after the French reports (1978 & 1984), although a satisfactory method of routinely measuring RCD levels in underground workings was not initially available. This was reduced to 1.5 mg/m$^3$ in the mid 1980’s and this limit is also currently used in Germany. In 2002 a reduction in RCD to 0.6 mg/m$^3$ was introduced in Quebec and in 2012, the limit was reduced to 0.4 mg/m$^3$ total carbon in Ontario.
Concerns have been expressed over the potential carcinogens associated with the particulate material emitted in diesel exhausts with the prospect of further reductions in threshold limit values. The possibility of diesel exhaust gases/particulates being carcinogenic was first seriously proposed in the late 1960's. In the intervening 45 years no definitive epidemiological evidence has been presented despite considerable research in this area.

Animal inhalation studies conducted on behalf of the National Institute for Occupational Safety and Health (NIOSH) in 1988 led to the recommendation that diesel exhaust may be regarded as a potential carcinogen. This was supported by the International Agency for Research on Cancer in 1989, which concluded that diesel engine exhaust is probably carcinogenic to humans.

None of this is particularly new or sufficiently definitive to justify a radical change in design values although it is always prudent to minimise any exposure to diesel exhaust gases and particulate. In the early 1980's it was recognised that there were as many mutagenic compounds in diesel fuel itself as in the diesel exhaust.

In 1998 the ACGIH initiated a notice of intended change for diesel exhaust particulate where the TWA limit was to be reduced to a value of 0.15 mg/m$^3$ for particulate less than 1.0 µm in size. More recently, diesel particulate matter (DPM) control limits of between 0.02 mg/m$^3$ and 0.4 mg/m$^3$ have been suggested with the limits based on what may be achievable with filters rather than on any epidemiological evidence that a hazard actually exists.

Although it may be possible to achieve this new limit by increasing the diesel exhaust gas dilution rates, it is more likely that diesel exhaust gas filters will be used. The ACGIH notice of intended change was not accepted and since 2004 there is no longer a limit for diesel particulate in the most recent ACGIH TLV’s and BEI’s handbooks.

In Australia, the NSW Minerals Council proposed a “best practice” guideline of 0.2 mg/m$^3$ for DPM and in Ontario Canada there is a proposal to adopt a DPM limit of 0.4 mg/m$^3$. In the 2008 NSW guidelines, it is stated that the epidemiological evidence for an association between diesel exhaust and lung cancer is still very weak and referred to a Mine Safety and Health Administration (MSHA) review published in 2001 that suggested a doubling of the risk of lung cancer for a 0.64 mg/m$^3$ (total carbon) exposure over 45 years exposure compared to that of unexposed miners.

The Australian Institute of Occupational Health has a position paper (May 2007) in respect of diesel particulate and contains the following statement in the summary: “Based on some of the animal and epidemiological studies, it is apparent that diesel particulate is a potential carcinogen. However, due to information deficiencies in the literature particularly regarding past exposure conditions, the AIOH has serious concerns as to the degree of potency being assigned to diesel particulates by some regulatory authorities.”

The results of an epidemiological study based on non-metal non-coal mines was published in 2012 by Silverman and Attfield. Although a statistically significant relationship between exposure to diesel engine exhaust and the incidence of lung cancer was claimed, there were limitations. The two main problems were that data on smoking and other confounders were not directly derived and, the retrospective dose assessment was incomplete and based on surveys undertaken between 1998 and 2001 and extrapolated back to the 1950’s.
A1.5 Acceptable noise levels

Background.
Acceptable noise levels depend on the criteria used for assessment and these may include; hearing impairment, signal or speech interference and, comfort levels. It is the first two criteria that are relevant in subsurface operations. However, in tunnelling the noise level of surface fans in built up areas must be considered.

The ear is sensitive to both level and frequency of the noise, the effect of frequency diminishing with increasing noise level. This is reflected by the three weighting networks where A-scale is suitable for sound pressure levels up to 55 dB, B-scale between 55 and 85 dB and C-scale for sound pressures over 85 dB. Weighting networks such as A, B and C-scales allow a single figure to be used to describe the complex range of frequencies that make up environmental noise.

Hearing loss or impairment depends on the level of noise, the duration of exposure and individual susceptibility. Acceptable equivalent continuous sound levels are usually given in terms of dBA despite normally being around 85 dB where the C-scale is designed to represent the sensitivity of the ear. The almost universal use of A-scale weighting is without basis except that it simplifies noise measurement and may result in values that are within normal measurement tolerances.

In a large epidemiological study undertaken in the UK in the 1970's, it was concluded that the B-scale weighting more nearly equates hearing loss than does the A-scale, and that no weighting overcompensates. A logical conclusion would be that, since C-scale weighting lies between B-scale weighting and no weighting, it would be more appropriate and would confirm the original purpose of the C-scale weighting network.

The practical consequence of using the A-scale weighting in hearing conservation is generally an error of the order of 2 dB. In tunnelling and mining, the frequency spectra tend to fall i.e. the mean of the sound levels in the 2000 and 4000 Hz bands are lower than the mean values in the 250 and 500 Hz bands. The use of A-scale weighting then leads to a small underestimate of the damaging effects of noise on hearing.

One definition of hearing loss is defined as the impairment in dB averaged over 1000, 2000 and 3000 Hz. A handicap may be where this average value is greater than 30 dB and disablement may be where it exceeds 50 dB. For a 35-year typical industrial exposure the proportion of people who may be affected is given in Table A1.5.

<table>
<thead>
<tr>
<th>Noise exposure (dBA)</th>
<th>None</th>
<th>85</th>
<th>90</th>
<th>95</th>
<th>100</th>
</tr>
</thead>
<tbody>
<tr>
<td>Over 30 dB loss</td>
<td>12</td>
<td>16</td>
<td>30</td>
<td>49</td>
<td>69</td>
</tr>
<tr>
<td>Over 50 dB loss</td>
<td>&lt; 5</td>
<td>9</td>
<td>17</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

It is relevant that without noise exposure, about 12% of persons older than 55 would have a hearing loss defined as a handicap and that exposure for 35 years at 85 dBA would cause this to rise to 16%. Noise levels of at least 90 to 95 dBA are necessary to result in a "disabling" hearing loss. Continuous equivalent noise levels of this intensity and where personnel do not use hearing protection are rare. This undoubtedly explains the very large discrepancy between postulated hearing loss disability claims and actual claims generally occurring in the mining industry world-wide.
Signal perception or speech interference occurs at noise levels above and below those concerned with hearing loss and usually occurs at frequencies between 500 and 2000 Hz at levels below hearing damage values. Permissible noise levels vary considerably and depend on the distance between the communicators. As a rough guide, communication is difficult at 0.5 m distance if the noise rating exceeds 80. For each doubling of distance, the noise rating would have to decrease by approximately 5 to maintain the same communication ability.

As mentioned previously, the A, B and C-scale weighting networks allow a single figure to be used to describe the noise level of an environment. They do not, however, yield any information as to the frequency of the high noise levels that may need controlling. To overcome this, noise rating curves have been developed. In a given situation, plotting the frequency spectrum on the family of curves allows the noise rating to be determined as the highest curve touched by the frequency spectrum.

**Recommendations.**
Where noise may cause speech or signal interference, the sound pressure levels should be reduced to limit the noise rating to between 70 and 80 dB. Where personnel may be exposed to noise for all or most of the shift, sound attenuation should be installed to ensure that the sound pressure level at the relevant location is 85 dBA or less. Where exposure to noise is intermittent and associated with travelling, sound attenuation should be installed to reduce the sound pressure level at the relevant location to less than 100 dBA.

Although the recommended design should aim for a maximum continuous equivalent sound level of less than 85 dBA, ear protection can be used to achieve this objective for intermittent sources (such as travelling past fans) providing that the maximum sound levels are less than the 100 dBA.
A1.6 Exposure standards and diesel exhaust dilution

General
Before discussing the appropriate exposure standards it is important to clarify two factors: -

1. Discussion or criticism of exposure standards does not imply that the project ventilation should be designed so that all the minimum exposure standards are only just met. Frequently, improved conditions in many other areas can be provided within the constraints of normal ventilation techniques and by following the best practice available for a particular criterion such as diesel exhaust dilution or minimum air velocity.

2. When dealing with most of the respirable pathogens encountered underground, it is not practical to monitor exposure of each person continuously (a possible exception to this is the monitoring of gamma radiation using film badges). Normally, relatively short duration measurements are taken and it is not practical to establish the actual time weighted average concentration. Consequently, if a short duration measurement exceeds the time weighted average concentration, an action is initiated as if personnel are over-exposed for the full working period. In practical terms, the time weighted average exposure limits are treated as short term excursion limits.

Proposed adjustments of the exposure standards usually have three features as follows: -

1. Introduction and application of the ALARA principle to atmospheric contaminants.
2. Introduction and application of reduction factors for extended shifts.
3. Adopting internal management action levels at some fraction of the exposure levels.

Oxides of nitrogen UK and EU change in limits
In the UK, Chemical Hazard Alert Notices (CHAN) 28 and 29 for nitrogen monoxide and nitrogen dioxide respectively both contain advice that the eight hour time TWA exposure limits for both gases should be reduced to 1 ppm. In 2003 the UK Health and Safety Commission had decided to withdraw the occupational exposure standards of 25 ppm for nitrogen monoxide and 3 ppm for nitrogen dioxide from the Occupational Exposure Standards (EH40/2002).

An interpretation is that although there is no statutory duty to meet the 1 ppm occupational exposure limits given in CHAN 28 and 29 for nitrogen monoxide and nitrogen dioxide, by not meeting the 1 ppm value, any employer may subsequently be held responsible for any medical consequences and required to pay compensation. By removing the maximum exposure limits from EH40/2002, the HSE is transferring the responsibility for any potential medical problems concerning the oxides of nitrogen if the exposure is greater than 1 ppm directly to the employer.

In UK law, EH40/2002 maximum exposure limits or occupational exposure limits would have been part of a defense against medical claims should they arise. The other part would have been the as low as reasonably achievable (ALARA) principle where evidence of a continuing improvement in conditions would need to be demonstrated.

The Chemical Hazard Alert Notices (CHAN) 28 and 29 refer to the UK Health and Safety Commission’s Working Group on the Assessment of Toxic Chemical (WATCH) who, on the basis of animal laboratory testing, believe that long term exposures to levels of nitrogen monoxide and nitrogen dioxide greater than 1 ppm may cause emphysema. The basis for this conclusion is not based on epidemiological data (see later) and does not conform to the EU Scientific Committee on Occupational Exposure Limits (SCOEL) recommendation of 0.2 ppm for nitrogen monoxide.

There are concerns with respect to the studies used to change the occupational limits and its
consequences on ventilation design criteria in that no other major country outside of the European Community appears to have taken note and adjusted the recommended or statutory nitrogen monoxide and nitrogen dioxide levels accordingly. Probably the most frequently quoted organization that makes recommendations on threshold limit values (TLV’s and equivalent to the UK COSHH occupational exposure limits) is the American Conference of Governmental Industrial Hygienists (ACGIH). The ACGIH limits are often specifically quoted in legislation (Ontario, Canada) or used as the basis for other countries occupational standards such as Australia NOHSC and in South Africa.

In the recent ACGIH editions the eight hour TLV-TWA remains as 25 ppm for nitric oxide (nitrogen monoxide) and 3 ppm for nitrogen dioxide. There is no notice of intended change for either of these gases nor are they under investigation with the possibility of future change. If the results of the studies referred to by WATCH were relevant to occupational exposures. It could be expected that the ACGIH would have reviewed them and taken action on changing the recommended occupational limits.

Possible confusion with pollution targets
Oxides of nitrogen are known to cause ozone problems caused by photochemical reactions in polluted atmospheres. There have been several protocols (Sofia in 1988 and Oslo in 1994) that attempt to deal with emissions and trans-boundary fluxes and the World Health Organization (WHO) also makes recommendations for exposure limits. It is not unusual for WHO recommendations to be about one tenth of any occupational limits (background to the difference between residential and occupational limits).

These limits would however apply at the boundaries to the project where the general population may be affected rather than as occupational limits for personnel working within the project boundaries.

A section from the 1988 International Labour Organisation (ILO) Encyclopaedia of Occupational Health and Safety is repeated to illustrate this point.

Air quality management aims at the preservation of environmental quality by prescribing the tolerated degree of pollution, leaving it to the local authorities and polluters to devise and implement actions to ensure that this degree of pollution will not be exceeded. An example of legislation within this approach is the adoption of ambient air quality standards based, very often, on air quality guidelines (WHO) for different pollutants; these are accepted maximum levels of pollutants (or indicators) in the target area (e.g. at ground level at a specified point in the community) and can be either primary or secondary standards.

Primary standards are the maximum levels consistent with an adequate safety margin and with the preservation of public health, and must be complied with within a specific time limit; secondary standards are those judged to be necessary for protection against known or anticipated adverse effects other than health hazards (mainly on vegetation) and must be complied “within a reasonable time”.

Air quality standards are short-, medium- or long-term values valid for 24 hours per day, 7 days per week, and for monthly, seasonal or annual exposure of all living subjects (including sensitive subgroups such as children, the elderly and the sick) as well as non-living objects; this is in contrast to maximum permissible levels for occupational exposure, which are for a partial weekly exposure (e.g. 8 hours per day, 5 days per week) of adult and supposedly healthy workers.

The information used as a rationale to change the occupational limits for the oxides of nitrogen was in response to air pollution concerns. The premise that all or a significant amount of
nitrogen monoxide converts to nitrogen dioxide in mining and tunneling is without foundation and wrong. In part this appears to have been accepted by WATCH and the UK Health and Safety Executive by changing the SCOEL value from 0.2 ppm to 1 ppm. Without evidence from an epidemiological study (slight or otherwise) the changed limits appear to be a bureaucratic anomaly and could be treated as such.

Background on animal studies
The Scientific Committee on Occupational Exposure Limits (SCOEL) had prepared a summary document (SEG/SUM/89D Jan 2001) on Nitrogen Monoxide (Nitric Oxide or NO) and this was distributed within the EU for discussion. From the SCOEL document, the recommended change in occupational limits to 0.2 ppm is heavily based on studies undertaken by Mercer at the US National Institute for Occupational Safety and Health (NIOSH). In August 2001, the background to the recommendations to reduce the occupational exposure limits of the oxides of nitrogen by SCOEL was reviewed by the European Fertilizer Manufacturers Association in an unpublished report that was critical of the recommended change.

Part of the US Health Effects Institute synopsis of the 1999 paper by Mercer is repeated:

“Dr Mercer found no increase in the number of holes in the alveolar scepta and no change in the thickness of individual alveolar wall compartments in animals exposed to 2 or 6 ppm NO compared with control animals exposed to air. These results differ from those of an earlier study, in which he and his colleagues used a lower concentration of NO (0.5 ppm). The discrepancy between findings of increased holes in the alveolar scepta in Dr Mercer’s previous study and no increase in the present study is an unaddressed issue and possible explanations should be explored. Analyses of the lung fluid showed no increase in chemical or cellular indicators of inflammation in animals exposed to NO. The study is one of the first to explore the toxicity of NO on lung structure and establishes a basis for future studies of the health effects of this air pollutant and therapeutic agent.”

A similar approach to changing occupational limits was originally taken in the USA with respect to diesel particulate where a 0.15 mg/m³ was proposed, not on the basis of epidemiological studies but on the what could be achieved with the removal of particulate from diesel exhausts using ceramic filters. The ACGIH notice of intended change was presented in 1998/99 and omitted in the later handbooks with no mention of an occupational limit for diesel particulate. It may be concluded that the basis for the reduction in diesel particulate limits were eventually found to be unsubstantiated.

Unfortunately it is unlikely that the European Community SCOEL has the mechanism to retreat from potentially spurious claims regarding occupational exposure levels to the oxides of nitrogen and these may then be promoted elsewhere. Most jurisdictions in Europe have remained with the original 3 ppm and 25 ppm limits for nitrogen dioxide and nitrogen monoxide including Spain and the recent Greek updating of regulations (July 2011) uses the higher values.

In the UK, the HSE suspended CHANs 28 and 29 recommending 1 ppm for both gases in 2009.
A1.7 Exposure levels for working extended shifts

General comments
In proposals for adjusting the exposure standards of extended shifts, two justifications are usually made which are of concern. The first is the claim that there is scientific consensus for an equivalent degree of protection for extended shifts and secondly, the exposure standards for inspirable or respirable dust have been established mainly on technological considerations rather than health effects. It is sometimes noted that where workers have a working day of longer than eight hours, the time weighted average exposure may need to be reduced by a suitable factor to ensure adequate worker protection. Further, such factors require specialist consideration and expert advice should be sought in the specification of modified exposure standards.

Application of the Brief and Scala model to carbon monoxide and other gases
The model recommended in many countries is the Brief and Scala model. Carbon monoxide is one of the main determinants for control of the exposure to airborne contaminants underground. Application of the Brief and Scala model for 12 hour shifts would result in the halving of the carbon monoxide time weighted average limit.

The Brief and Scala adjustment is:-

\[
\text{Adj. TWA} = \frac{8 \times (24 - h) \times (8 \text{ hour TWA})}{16 \times h}
\]

If the 8 hour TWA for carbon monoxide is 50 ppm, and the shift length h is 12 hours, the adjusted 12 hour TWA is 25 ppm.

The two factors to be considered when using adjusted levels for extended shifts are the effect of the longer exposure and the shorter time available for the body to recover. The chronic effects of carbon monoxide are related to a biological index or carboxy-haemoglobin (HbCO) concentration and the relationship between exposure level and duration of exposure is given by the Coburn model illustrated in Figure A1.1.

With respect to the effect of longer exposure periods for continuous exposures at carbon monoxide levels of less than 50 ppm, the Coburn model shows no increase in HbCO levels after 8 hours exposure i.e. the HbCO level will not increase when the shift length is longer than 8 hours. The recovery time or rate at which carbon monoxide is excreted from the body is approximately 3 to 4 hours for the HbCO level to reduce by one half. If the worst case is assumed i.e. the excretion rate at low levels of HbCO is exponential, after 12 hours the level will be about one tenth and after 16 hours, about one twentieth. At these very low levels it is most unlikely that any discernible difference could be considered as an increase in risk.

Relative to the acceptable concentration for exposure to carbon monoxide of 50 ppm applied generally prior to 1995 and currently in South Africa, the extended shift limit using the Brief and Scala model is 50% of this at 25 ppm. There is no epidemiological evidence to demonstrate or even suggest that any change in the 50 ppm exposure limit is justified and not to half of what was previously considered to be a "safe" value. It is concluded that there is no justification for adjusting the time weighted average for occupation exposures to carbon monoxide when the shift length exceeds 8 hours.

The other gases that may be encountered in Namib North mine are carbon dioxide, nitric oxide, nitrogen dioxide and sulphur dioxide. Any problem with carbon dioxide is concerned with asphyxiatiation and reduced limits for longer shifts are not relevant. The main biological health effect of the remaining gases is as an irritant and similarly there is no evidence for accepting lower limits for longer shifts.
Figure A1.1 – Coburn model carboxyhaemoglobin levels for heavy work
APPENDIX 2 – DIESEL DECLINE HAULAGE VENTILATION REQUIREMENTS

In section 2.2, reference was made to a ventilation rate of 7.5 m³/s per Mtpa.km in relation to diesel powered truck haulage through a decline. This is typical for the equipment currently available and has been validated in practice using diesel fuel consumptions.

A2.1 Truck power, payload and speed

Truck speed is a function of grade, rolling resistance, power, payload and total mass and varies between relatively narrow limits. For haulage up a 1:8 decline the speed can be obtained from:

\[
\text{Speed (km/h)} = \frac{\text{rated power (kW)}}{(\text{factor } \times \text{payload})} - \text{constant.}
\]

Typical values of the factor and constant are obtained from the following table.

<table>
<thead>
<tr>
<th>Factor</th>
<th>Constant</th>
</tr>
</thead>
<tbody>
<tr>
<td>Maximum</td>
<td>0.45</td>
</tr>
<tr>
<td>Average</td>
<td>0.55</td>
</tr>
<tr>
<td>Minimum</td>
<td>0.65</td>
</tr>
</tbody>
</table>

The speed is that up the decline with the truck fully loaded and operating at 100% engine power. The speed down the decline is taken as double the speed up the decline and an engine power of 50% which is used mainly for retardation. A rolling resistance of 2.5% is assumed and a maximum downgrade speed of 25 km/h.

A2.2 Specific ventilation rate - general

Assume a 1000 m depth, 1:8 decline, Toro 40 truck with a rated power of 354 kW and a payload of 40 tonne. Using the average values given in Table A1.1, the truck speed up the decline is 354/(0.55 x 40) – 7.7 = 8.4 km/h. The decline length is 8000 m (1000 m depth and 1:8 grade) and the time to haul 40 tonnes up the decline is 3429 s. The return time is 1714 s for a cycle time in the decline of 5143 s. The weighted power is 354 x (3429 x 1.0 + 1714 x 0.5)/5143 = 295 kW.

The number of cycles per year is (3600 x 24 x 365)/5143 = 6132 and the amount of rock hauled is 6132 x 40 = 245 280 tonnes or 0.2453 Mtpa. The haul distance is 8000 m or 8 km and the truck performance is therefore 1.962 Mtpa.km.

The weighted power is 295 kW and the power for 1.0 Mtpa.km is 295/1.962 = 150 kW. Using a specific dilution rate of 0.05 m³/s per kW results in a specific ventilation rate for the haulage of 150 x 0.05 = 7.5 m³/s per Mtpa.km.

A2.3 Specific ventilation rate – AD55B

For the AD55B truck, instead of using a generalised speed relationship, the speed up the decline can be obtained from a Gradeability/Speed/Rimpull graph. For a 1:7 gradient and a 2.5% rolling resistance, the fully loaded truck speed is 9.7 km/h. The cycle time is 3897 s (depth 1000 m) and the weighted power is 483 kW (rated power is 579 kW).

The annual tonnes hauled is 445 080 or 3.116 Mtpa.km. Based on the weighted power of 483 kW, the specific ventilation rate is 155 x 0.05 = 7.75 m³/s per Mtpa.km.
APPENDIX A3 - ECONOMIC AIRWAY ANALYSIS

To guide in the selection of airway sizes and to determine the approximate air handling capacities of existing airways, the most economic airway sizes are obtained using a net present value and rate of return method that incorporates adjustments for both tax and depreciation with savings.

It is rare that the air carrying capacity of an airway is constant over the life of the mine and consequently the effects of non-optimum conditions also needed to be determined. Although there may be an optimum airway size where the rate of return is greatest, there is also a range of values where the minimum financial criteria for the project are met.

A3.1 Airway friction factors and cost data.

The surface roughness of underground airways generally falls into two broad categories; drill and blast surfaces or bored/concrete lined surfaces. The method used to determine the Atkinson friction factor is based on equivalent sand roughness and the Colebrooke and White empirical relationship.

For drill and blast airways with an average absolute roughness height of 0.15 m and a hydraulic diameter of 3.5 m, the frictional coefficient $\lambda$ is 0.067 resulting in an Atkinson friction factor of 0.010 Ns$^2$/m$^4$. For a raise bored or concrete lined airway with an average roughness height of 0.01 m and a diameter of 4.0 m, the frictional coefficient lambda is 0.025 resulting in an Atkinson friction factor of 0.0037 Ns$^2$/m$^4$.

The cost of the airways in $/m can be obtained from:

\[
\text{Drill and blast} \quad C_{r} + C_{v} A \\
\text{Underground RBH} \quad C_{r} + C_{v} D^4
\]

where

- $A$ = cross sectional area
- $D$ = raise diameter
- $C_{r}$ = airway fixed cost component
- $C_{v}$ = airway variable cost component

A3.2 Financial criteria and power requirements.

The financial variables are defined as:

- Life - n years
- Interest - i %
- Depreciation - linear or with profits
- Tax - t, %

The cumulative present value factor

\[
cpv = \frac{100 \left( \left[ 1 + \frac{i}{100} \right]^n - 1 \right)}{i \left( 1 + \frac{i}{100} \right)^n}
\]
using: \[ Pd = \frac{KCLQ^2}{A^3} \]  and  \[ \text{power} = \frac{PdQ}{1000n_f} \]

where \( Pd \) = pressure loss (Pa)
\( K \) = Atkinson friction factor (\( \text{Ns}^2/\text{m}^4 \))
\( C \) = airway circumference (m)
\( L \) = airway length (m)
\( Q \) = Air quantity (\( \text{m}^3/\text{s} \))
\( A \) = airway cross sectional area (\( \text{m}^2 \))
\( n_f \) = fan efficiency

therefore: \[ \text{power} = fKQ^3D^{-5}n_f^{-1} \]

For circular airways \( f = 0.006485 \) and \( D = \text{diameter} \)
For other airways \( f = 0.004 \) and \( D = \sqrt{\text{area}} \)

### A3.3 Financial evaluation - linear depreciation.

The cost criteria relating to the fans and airways used in the optimisation are defined as follows:

- Power cost \(- C_p \) (\$/kWh)
- Maintenance \(- C_m \) (proportion of \( C_p \))
- Fan cost \(- C_f \) (\$/kW)
- Airway cost \(- C_{rs} + C_r D^3 \)

Annual costs = 8760 \( fKQ^3D^{-5}n_f^{-1}C_pC_m \)

Initial costs = \( fKQ^3D^{-5}n_f^{-1}C_f + C_{rs} + C_r D^3 \)

Defining the difference in operating and capital costs as:

\[ \text{Dop} = \text{Baseop} - 8760 fKQ^3D^{-5}n_f^{-1}C_pC_m \]
\[ \text{Dcap} = fKQ^3D^{-5}n_f^{-1}C_f + C_{rs} + C_r D^3 \]

Assuming a linear depreciation over the life of the airway, the net present value after \( n \) years

\[ \text{NPV}_n = cpv \left[ 1 - \frac{t_x}{100} \right] \left( \frac{\text{Dop} - \text{Dcap}}{n} \right) + \frac{\text{Dcap}}{n} - \text{Dcap} \]

Baseop and Basecap are the costs used for comparison and are either determined by limiting constraints such as air velocity or meeting minimum financial criteria.

Defining: \( f_1 = cpv(1 - t_x/100) \) and \( f_2 = f_1/n - cpv/n + 1 \)

\[ \text{NPV}_n = f_1 \text{Baseop} - f_1 8760 fKQ^3D^{-5}n_f^{-1}C_pC_m - f_2 \text{Basecap} - f_2 fKQ^3D^{-5}n_f^{-1}C_f \]
\[ - f_2 C_{rs} - f_2 C_r D^3 \]

Differentiating and equating to zero to obtain the maximum NPV:

\[ 0 = 5(f_1 8760 fKQ^3D^{-6}n_f^{-1}C_pC_m + f_2 fKQ^3D^{-6}n_f^{-1}C_f) - x f_2 C_r D^{x-1} \]

Dividing by \( x f_2 C_r D^{x-1} \) and re-arranging
A3.4 Minimum and maximum airway sizes.

For a given air quantity, as the airway size is reduced, the initial airway cost decreases and the pressure loss increases. This results in higher initial fan costs that may be larger than the reduction in airway cost. Differentiating the total initial costs of the airway and the fan cost and equating this to zero obtain the minimum airway size: -

$$D_{\text{min}} = \frac{C_f fKQ^3n_f^{-1}}{\left(\frac{xf_2C_r}{5}\right)^{1-x}^{1+x}}$$

As the airway size is increased above the optimum size, for a given air quantity, the pressure loss will decrease and the power and fan costs will also decrease. The initial airway cost will normally increase at a greater rate and will reduce the net present value. The maximum airway size is that where the net present value is zero i.e. the size of airway that will result in the required interest rate. This value is obtained by using the minimum airway size to determine the base costs and an iterative method to establish when the net present value is zero.

A3.5 Financial evaluation - depreciation with savings.

When depreciating the capital against the savings in operating costs (potential profits), the net present value function is not uniform and the solution is obtained incrementally by considering each year separately. The minimum airway size is used as the starting point for the analysis and provides the base values. For each year the net present value factor is obtained from: -

$$pvf = \frac{1}{\left(1 + \frac{i}{100}\right)^n}$$

The residual capital D_{cap} for the year in question is equal to the residual value for the previous year less the difference in operating costs D_{op}. If this is a positive value, the net present value for the year is obtained from: -

$$\text{npv} = pvf \cdot D_{op}$$

If the residual becomes negative during the year in question, the net present value for the year is obtained from:
If the residual was already negative, the net present value for the year is obtained from:

\[ npv = pvf \left[ \left( 1 - \frac{t_s}{100} \right) Dop - Dcap \right] + Dcap \]

For each airway size, the net present values for the individual years were summed, reduced by the difference in initial costs and the maximum value obtained (the optimum airway size). As before, the maximum airway size was that where the cumulative net present value was equal to zero.
Appendix C  Risk Assessment Report
<table>
<thead>
<tr>
<th>Reference</th>
<th>Description</th>
<th>Cause/Description</th>
<th>Evaluation</th>
<th>Risk Management Planning</th>
<th>Actions</th>
<th>Notes</th>
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<td>Resource Estimation</td>
<td><strong>Probability</strong></td>
<td>1.00</td>
<td>Resource Estimation</td>
<td>NRR</td>
<td>Snowden</td>
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<td>Resource Estimation</td>
<td><strong>Consequences</strong></td>
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<td>Resource Estimation</td>
<td>NRR</td>
<td>Snowden</td>
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**Description**
- Resource Estimation:
  - **Probability**: 1.00
  - **Consequences**: 4.00
  - **Risk Value**: 1.00
- Operations:
  - **Probability**: 1.00
  - **Consequences**: 4.00
  - **Risk Value**: 1.00
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<th>Risk Management Planning</th>
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<td>Risk 3/03/2014 HTM</td>
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<td>Risk 3/03/2014 HTM</td>
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<td>Risk 7/03/2014</td>
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<td>Risk 3/03/2014 HTM</td>
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**Sample Entry:**

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<th>Date Raised</th>
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<th>Title &amp; Description</th>
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<th>Consequence</th>
<th>Risk Value</th>
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<tr>
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<td>Collapse of crown pillar</td>
<td>3.08</td>
<td>Risk</td>
<td>248A</td>
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<tr>
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<td>MS/RM</td>
<td>Damage to equipment and injury to personnel as a result of</td>
<td>3.09</td>
<td>Risk</td>
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<td>7/03/2014</td>
<td>MS/RM</td>
<td>Exposed ore faces, poor development rates or excessive</td>
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<td>MS/RM</td>
<td>Power failures</td>
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<td>MS/RM</td>
<td>Samples not representative</td>
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<td>7/03/2014</td>
<td>HTM</td>
<td>Testwork incomplete at time of finalising</td>
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<td>7/03/2014</td>
<td>HTM</td>
<td>Fe suppression not effective</td>
<td>4.03</td>
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<td>7/03/2014</td>
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<td>Lower than expected flotation recovery</td>
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<td>Risk</td>
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<tr>
<td>5/03/2014</td>
<td>HTM</td>
<td>Rock hardness - Impact on crusher</td>
<td>4.05</td>
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<tr>
<td>7/03/2014</td>
<td>HTM</td>
<td>Equipment failure - lead time for</td>
<td>4.06</td>
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<td>5/03/2014</td>
<td>HTM</td>
<td>Water balance issue</td>
<td>4.07</td>
<td>Risk</td>
<td>236</td>
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</table>

**Mitigation required:**

- **Risk Management Planning Notes:**
  - **Issue:**
    - Risk, Opp or
  - **Date Raised:**
    - 7/03/2014
  - **Raised By:**
    - MS/RM
  - **Title & Description:**
    - Collapse of crown pillar, can be devastating, one way traffic, potential of being on duty in a bit of right
  - **Probability:**
    - 3.08
  - **Consequence:**
    - Risk
  - **Risk Value:**
    - 248A
  - **Mitigation required:**
    - Stopes near surface may collapse the crown pillar resulting in continual monitoring.
  - **When Owner 1 Owner 2:**
    - Ref

**Reference:**

- **Risk Management Planning Notes**
- **Event in:**
- **Risk Management Planning:**
- **Note:**
  - **Ref**
  - **Action**
  - **Resource**
  - **Date**
  - **Operation NRR Snowden**
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<tbody>
<tr>
<td>4.13 Risk 7/03/2014 MS</td>
<td>Over specify filters initially, but this may impact negatively on Capex and Opex.</td>
<td>3.12</td>
<td>12</td>
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<td>12</td>
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<td>4.14 Risk 7/03/2014 MS</td>
<td>Filtration tests can only be conducted once saleable grade concentrates have been produced.</td>
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<td>103</td>
<td>103</td>
<td>103</td>
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<td>4.15 Risk 1/04/2014 HTM</td>
<td>If lightning protection is required, client to include contingency and have engineer who is registered with the Engineering Council of Namibia sign-off.</td>
<td>4.16</td>
<td>20</td>
<td>20</td>
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<tr>
<td>5.02 Risk DK</td>
<td>Piping/pipework Piping to the sewerage plant from the change-houses - Use fire extinguishers in those areas.</td>
<td>2.5</td>
<td>10</td>
<td>10</td>
<td>10</td>
<td>10</td>
<td>10</td>
<td>10</td>
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<tr>
<td>5.03 Risk WR</td>
<td>Topographical Survey A survey of the plant footprint is required after clearance/levelling of the site to be done.</td>
<td>5.16</td>
<td>20</td>
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<td>20</td>
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<tr>
<td>5.04 Risk DK</td>
<td>Electrical grid tie-in and overhead lines No capital provision has been made for overhead lines to connect to the grid.</td>
<td>5.16</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>5.05 Risk DK</td>
<td>Fire water reticulation There has been no provision made for fire water distribution to the plant.</td>
<td>5.16</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>5.06 Risk DK</td>
<td>Testwork incomplete within schedule Assumptions based on testwork results from previous scoping studies proved to be inaccurate.</td>
<td>5.16</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>5.07 Risk WR</td>
<td>Availability of water to the processing facility The main supply to Rossing has lots of spare capacity to cover future demands.</td>
<td>5.16</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
</tr>
<tr>
<td>5.11 Risk WR</td>
<td>Site recovery from tailings pond 26/06/2014 PJP</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>5.12 Risk WR</td>
<td>Critical and material contracts unsigned or in draft form</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>5.13 Risk WR</td>
<td>Critical and material contracts unsigned or in draft form</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>5.14 Risk PvdM</td>
<td>Availability of local labour If there is a requirement for using local labour during execution of the project, North River could be competing with the Husab project for local skills/labour availability.</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>5.15 Risk WS</td>
<td>Availability of labour on site</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>5.16 Risk WS</td>
<td>Site preparation for the project</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>6.02 Risk MS</td>
<td>Samples used for testwork may not be representative of the ore body.</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>6.03 Risk MS</td>
<td>Zinc recovery from Tailings may not be achieved.</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>6.04 Risk MS</td>
<td>availability of spares (either from site or from external sources), transportation and delivery to Project site.</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>6.05 Risk WS</td>
<td>Critical and material contracts unsigned or in draft form</td>
<td>6.03</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>6.06 Risk WS</td>
<td>Critical and material contracts unsigned or in draft form</td>
<td>6.03</td>
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<td>395</td>
<td>395</td>
<td>395</td>
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<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
<td>395</td>
</tr>
<tr>
<td>Date Raised</td>
<td>Raised By</td>
<td>Title &amp; Description</td>
<td>Cause/description</td>
<td>Probability</td>
<td>Consequence</td>
<td>Risk Value</td>
<td>Mitigation required</td>
<td>When</td>
<td>Owner 1</td>
</tr>
<tr>
<td>-------------</td>
<td>-----------</td>
<td>---------------------</td>
<td>-------------------</td>
<td>-------------</td>
<td>-------------</td>
<td>------------</td>
<td>---------------------</td>
<td>------</td>
<td>--------</td>
</tr>
<tr>
<td>26/06/2014</td>
<td>PJP</td>
<td>Poor roadway conditions and layout</td>
<td>Poor maintenance and increasing traffic, thus increasing transport risk and affecting inbound/outbound delivery schedules.</td>
<td>3.33</td>
<td>Resource Estimation</td>
<td>2.09</td>
<td>Ongoing monitoring of road; frequent liaison with municipality</td>
<td>1.00</td>
<td>Resource</td>
</tr>
<tr>
<td>26/06/2014</td>
<td>PJP</td>
<td>Quality of the diesel supplied by contractor</td>
<td>Quality of the diesel supplied by contractor is poor, affecting diesel consumption and increasing operational and production costs.</td>
<td>2.50</td>
<td>Resource Estimation</td>
<td>1.71</td>
<td>Purchase diesel quality kit</td>
<td>6.04</td>
<td>NNR</td>
</tr>
<tr>
<td>26/06/2014</td>
<td>PJP</td>
<td>Ineffective response to underground emergency</td>
<td>Ineffective emergency systems underground</td>
<td>2.00</td>
<td>Resource Estimation</td>
<td>6.02</td>
<td>Adequate investigations and design studies. Develop and implement fire suppression systems and pressure reducing stations are to be enclosed to prevent damage</td>
<td>6.06</td>
<td>NNR</td>
</tr>
<tr>
<td>4/03/2014</td>
<td>R Fry</td>
<td>Adverse geotechnical conditions. Conditions not as anticipated from investigations. Inadequate investigations. Inappropriate mine and support design. Poor quality installation of support.</td>
<td>Adverse geotechnical conditions. Conditions not as anticipated from investigations. Inadequate investigations and/or assessment. Inappropriate mine design. Poor quality installation of support. Poor geotechnical management plan.</td>
<td>5.00</td>
<td>Excessive Dilution</td>
<td>6.00</td>
<td>Undertake detailed investigations into unstable geotechnical conditions. Develop and implement geotechnical management plan.</td>
<td>7.00</td>
<td>NRR</td>
</tr>
<tr>
<td>03/03/2014</td>
<td>Prime</td>
<td>Not yet informed of authority decision. ESIA was submitted on 17/12/2013</td>
<td>Not yet informed of authority decision. ESIA was submitted on 17/12/2013</td>
<td>5.00</td>
<td>Social &amp; Environmental</td>
<td>6.00</td>
<td>Follow up with authorities. Amend ESIA if necessary and re-submit</td>
<td>8.00</td>
<td>NRR</td>
</tr>
<tr>
<td>03/03/2014</td>
<td>Prime</td>
<td>Inadequate planning for rehabilitation and closure. Significant contamination site.</td>
<td>Inadequate planning for rehabilitation and closure. Significant contamination site. Unknown actions to be taken to rehabilitate the tailings dams or prevent spread of tailings dust.</td>
<td>5.00</td>
<td>Social &amp; Environmental</td>
<td>6.00</td>
<td>Undertake detailed investigations into rehabilitation and closure options. Adequate construction QA/QC. Develop and implement TSF management plan.</td>
<td>8.00</td>
<td>NRR</td>
</tr>
<tr>
<td>03/03/2014</td>
<td>Prime</td>
<td>Obtain missing page and assess implementation of obligations in Environmental Contract</td>
<td>Missing page from signed Environmental Contract. Insufficient funds made available for closure. No financial provision for closure has been determined. Rough estimate is at ZAR 15-20 Million.</td>
<td>5.00</td>
<td>Social &amp; Environmental</td>
<td>6.00</td>
<td>Finalise engineering designs and investigations into closure options and determine required financial provision</td>
<td>8.00</td>
<td>NRR</td>
</tr>
<tr>
<td>18/03/2014</td>
<td>NRR_D</td>
<td>Insufficient funds made available for closure. No financial provision for closure has been determined. Rough estimate is at ZAR 15-20 Million.</td>
<td>Insufficient funds made available for closure. No financial provision for closure has been determined. Rough estimate is at ZAR 15-20 Million.</td>
<td>5.00</td>
<td>Social &amp; Environmental</td>
<td>6.00</td>
<td>Finalise engineering designs and investigations into closure options and determine required financial provision</td>
<td>9.00</td>
<td>NRR</td>
</tr>
<tr>
<td>18/03/2014</td>
<td>NRR_D</td>
<td>Obtain missing page and assess implementation of obligations in Environmental Contract</td>
<td>Obtain missing page and assess implementation of obligations in Environmental Contract</td>
<td>5.00</td>
<td>Social &amp; Environmental</td>
<td>6.00</td>
<td>Follow up with authorities. Amend ESIA if necessary and re-submit</td>
<td>9.00</td>
<td>NRR</td>
</tr>
<tr>
<td>26/06/2014</td>
<td>JE</td>
<td>Significant increase in costs of production</td>
<td>Macro-economic factors</td>
<td>2.50</td>
<td>Cost of Production</td>
<td>3.00</td>
<td>Lower production and production cost</td>
<td>10.01</td>
<td>NRR</td>
</tr>
</tbody>
</table>
Appendix D   Environmental Review
North River Resources PLC

Environmental and Social Review

Project No. J2178

Namib Lead Zinc Project

February 2014
This report has been prepared by Prime Resources (Pty) Ltd (Prime) on behalf of Snowden Mining Consultants (Snowden) for North River Resources PLC.

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Prepared By
Gené Main
MSc (Botany), BSc (Hons) Environmental Science
Senior Environmental Scientist .............................................

Reviewed By
Peter Theron
Pr Eng BSc (Eng) Civil; GDE Environmental Engineering, Tailings & Waste
Managing Director, Environmental Engineer
..............................................................................................

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North River Resources PLC: 1 electronic
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1 Introduction

1.1 Background

North River Resources (Pty) Ltd (NRR) holds 100% of the Namib Lead Zinc Project (NLZM), in Swakopmund, Namibia. The NLZM is centred on the previously operational (closed in 1992) underground Namib Lead Mine. NRR currently has in place an Exclusive Prospecting Licence (EPL), and is due to submit a mining licence application by 17 April 2014. An Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) was compiled by Colin Christian & Associates CC (CCA) as supporting documentation for the mining licence application, and submitted to the Ministry of Environment and Tourism (MET) on 17 December 2013.

1.2 Objectives

Prime Resources was appointed to undertake a desktop review of the current status of the NLZM from an environmental and social perspective. In order to address this, all currently available documentation was reviewed. This includes the final Environmental Impact Assessment (EIA) (dated 13 December 2013) as compiled by CCA, and the associated specialist studies:


- Burke, A. June 2012. Flora baseline for the Namib Lead Mining Prospect (EPL 2902) in the central Namib Desert. [Envirosience].

- Irish, J. 27 August 2012. Biodiversity scoping study for EPL 2902. [Biodata Consultancy].

- Kinahan, J. 9 November 2013. Archaeological field survey and assessment of EPL 2902 (Namib Lead), Erongo Region. [Quaternary Research Services].
2 Brief Project Description

2.1 Objectives

This review was undertaken of the available documentation at the time (as of 17 December 2013). The documentation included the final Environmental Impact Assessment (EIA), Environmental Management Plan (EMP) (dated 13 December 2013) and appendices, as submitted to MET on 17 December 2013.

The review provides an independent opinion on the environmental and social aspects of the project, including the contents of the EIA and EMP, identifying any fatal flaws or risks to the project, commenting on compliance with national (Namibian) legislation, and recommending actions to address any obvious concerns.

2.2 Introduction

NRR is proposing to re-open a previously operational underground lead mine, which closed in 1992. The intention is to mine lead and zinc (and small amounts of silver) for export. The old underground mine is to be extended both vertically and horizontally, and additional shafts from the surface will be required. Exploration drilling has focussed mainly on targets adjacent to and below the old mine workings. Drilling has been conducted both from surface and underground.

NRR has proposed to develop the mine in two stages:

Stage 1 will include advanced exploration (mostly underground), and no processing or production. Access drives need to be developed to facilitate drilling from underground positions. Several new exploration targets have also been identified, which are close to but unconnected to the old mine. Drilling of these targets from the surface is proposed until October 2014.

Stage 2 of the mine development will occur once the target ore resource has been proven. A new processing plant will be set up adjacent to the old mine to produce lead and zinc concentrates. This is proposed from October 2014 to December 2021.

Rehabilitation and closure will occur after Stage 2. It is envisaged that waste rock may be disposed of underground in old workings during Stages 1 and 2. Surface rehabilitation will occur after mining. Closure is anticipated to occur between January and June 2022.

2.3 Project location and layout

The Exclusive Prospecting Licence (EPL 2902) is situated in the recently proclaimed (2010) Dorob National Park, just to the west of Rössing Mountain, and covers an area of 4,523 ha (Figure 2.1). It lies between Swakopmund (23 km) and Arandis (34 km) (Figure 2.2).
2.4 Proposed mine plan

A target production rate of 250,000 tonnes per annum is envisaged by NRR. This translates to a seven year Life of Mine. It is proposed that the products will be transported in containers by road to the Port of Walvis Bay for export. This will require two trucks per day, each carrying two 20 foot (6.1 m) containers, per trip.
Tailings material from the “old” tailings dam will be reprocessed and disposed of onto the “existing” tailings dam, as developed in the mid-1990s. Some of the waste rock from new development will be disposed of in existing voids underground to minimise haulage costs and provide stabilising support where needed. Waste rock that cannot be disposed of underground will be hauled to the surface and dumped adjacent to the old mine. Tailings produced by new processing will remain on the surface.

Water will be supplied by a new 110 mm pipeline on an existing route, from an offtake on the Swakop–Rössing Mine pipeline. A surplus is also available from the desalination plant of a neighbouring mine, Areva. An existing borehole on site can be pumped to provide a supply of non-potable saline water, suitable for drilling activities. Power will be supplied via an existing line from the national grid. If insufficient power is available, diesel generators will be installed to supply additional power. ErongoRED is only able to supply 500 kW, which is considered sufficient for Stage 1 only.

No company housing will be provided. Staff will be transported by bus to and from Swakopmund and Walvis Bay to support the scheduled shifts.

The new mine and infrastructure will be developed within the existing footprint. Figure 2.3 shows the proposed mine area in more detail, showing both the existing and proposed infrastructure. Dust fallout from the mine area and tailings dam stretches for over 1 km to the south-west. No rehabilitation was conducted after the previous mine closed in 1992.

Figure 2.3 Existing and proposed mine infrastructure

---

1 Two tailings dams were constructed during the previous mining activities. The “existing” tailings dam will form part of the footprint of the proposed tailings dam.
3 Legal Framework

3.1 National Legislation

3.1.1 The Constitution

The environment is constitutionally protected in Namibia. The Constitution is the supreme law of Namibia and all other law is subordinate and subject to the constitutional provisions and principles, including international law. Two important environmental clauses of the Namibian Constitution are:

Article 91(c) - defines the functions of the Ombudsman to include “the duty to investigate complaints concerning the over-utilisation of living natural resources, the irrational exploitation of non-renewable resources, the degradation and destruction of ecosystems and failure to protect the beauty and character of Namibia”, and

Article 95(l) - commits the state to actively promoting and maintaining the welfare of the people by adopting policies aimed at the “maintenance of ecosystems, essential ecological processes and biological diversity of Namibia and utilisation of living natural resources on a sustainable basis for the benefit of all Namibians, both present and future”.

3.1.2 Environment Management Act (No. 7 of 2007) and Environmental Impact Assessment Regulations (GN4878 of 2012)

The Environmental Management Act (EMA) (2007) establishes the requirements for Environmental Impact Assessments (EIAs) and Environmental Management Plans (EMPs). The EMA requires, among others, that assessments must be undertaken for activities which may have a significant effect on the environment or the use of natural resources; that natural and cultural heritage must be protected; that effective waste management must be promoted; and that persons who cause damage to the environment must pay the costs associated with rehabilitation.

The EMA Regulations (GN 4878) (2012) list activities that require an EIA and may not be carried out without an Environmental Clearance Certificate (ECC) issued by the Environmental Commissioner in the Ministry of Environment and Tourism (MET). Relevant listed activities are noted in Table 3.1.
Table 3.1  Listed activities potentially associated with the proposed Namib Lead Zinc Project

<table>
<thead>
<tr>
<th>Activity No.</th>
<th>Category</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Energy Transmission and Storage</td>
<td>b) The transmission and supply of electricity</td>
</tr>
<tr>
<td>2.1</td>
<td>Waste Management, Treatment, Handling and Disposal</td>
<td>The construction of facilities for waste sites, treatment of waste and disposal of waste</td>
</tr>
<tr>
<td>2.2</td>
<td>Waste Management, Treatment, Handling and Disposal</td>
<td>Any activity entailing a scheduled process in terms of the Atmospheric Pollution Prevention Ordinance, 1976</td>
</tr>
<tr>
<td>3.1</td>
<td>Mining and Quarrying</td>
<td>The construction of facilities for any process or activities which requires a licence, right or other form of authorisation, and the renewal of a licence, right or other form of authorisation, in terms of the Minerals Prospecting and Mining Act, 1992</td>
</tr>
<tr>
<td>5.1</td>
<td>Land Use and Development</td>
<td>Change of land use / rezoning, from nature conservation to any other land use</td>
</tr>
<tr>
<td>8.1</td>
<td>Water Resource Developments</td>
<td>The abstraction of groundwater or surface water for industrial or commercial use</td>
</tr>
<tr>
<td>8.4</td>
<td>Water Resource Developments</td>
<td>Construction of canals or channels including the diversion of the normal flow of water in a riverbed</td>
</tr>
<tr>
<td>8.5</td>
<td>Water Resource Developments</td>
<td>Construction of dams, reservoirs or weirs</td>
</tr>
<tr>
<td>8.6</td>
<td>Water Resource Developments</td>
<td>Construction of industrial and domestic wastewater treatment plants and related pipeline systems</td>
</tr>
<tr>
<td>9.1</td>
<td>Hazardous Substance Treatment, Handling and Storage</td>
<td>Manufacture, storage, handling or processing of any substance defined in the Hazardous Substances Ordinance, 1974</td>
</tr>
<tr>
<td>10.1</td>
<td>Infrastructure</td>
<td>The construction of and j) communication masts and lightning conductors</td>
</tr>
</tbody>
</table>

3.1.3 Minerals (Prospecting and Mining) Act (No. 33 of 1992)

The Minerals Act controls the licences for all prospecting and mining in Namibia. NRR currently has in place an EPL for the site. Section 5.8 of the latest EPL renewal, dated 28 May 2012, states that the last possible renewal has been granted, and the EPL is valid until 17 April 2014. NRR has reportedly met with the Permanent Secretary who indicated the possibility for further renewals at the discretion of the Minister.

Prospecting is considered a listed activity in terms of the EMA (above), and requires an Environmental Clearance Certificate (ECC). The ECC, dated 31 January 2013, requires that the Environmental Contract entered into during the previous tenure must be adhered to. The ECC dated 13 June 2001 contains the relevant information forming the Environmental Contract. The conditions relate to hazardous materials, waste management, water use, and protection of fauna and flora\(^2\). There is no evidence to suggest that these conditions have not been adhered to. There is however, further reference to the provision of Environmental Reports to the Ministry of Mines and Energy (MME), but full details are unknown. It is unconfirmed whether such reports were submitted as required.

---

\(^2\) At least one page was missing from the 2001 ECC obtained for the review. Full obligations in terms of this Environmental Contract can therefore not be established.
The application for a mining licence must address, among others, the baseline condition of the site, an indication of impacts potentially caused by the project, and mitigation and management measures. These items have been addressed in the EIA/EMP, which was submitted to MET on 17 December 2013. It is expected that authorisation (an ECC) will be obtained on the basis of this submission, and this will be submitted in support of the mining licence application.

General provisions of the Minerals Act require that the holder of a prospecting or mining licence is liable for the costs of rehabilitation and closure. No financial provision for closure has yet been determined because much of the design and engineering work is still to occur. The EIA recommends the use of the South African Minerals and Petroleum Resources Development Act (MPRDA)’s guidelines to determine the likely cost of rehabilitation and closure. The EIA further notes that NRR has indicated support of the concept of establishing a trust fund specifically for rehabilitation and closure.

3.1.4 Nature Conservation Ordinance (No. 4 of 1975, as amended)

The NLZM is situated within the Dorob National Park. The ordinance lists protected plants, and requires a permit to destroy or move/remove certain species. The EIA notes that one sensitive species, *Lithops ruschiorum*, occurs near the footprint of the proposed activities, and will not be affected by mining. It does however note that prospecting activities may impact on these populations, and requires raising of awareness and monitoring these populations. If impacts are noted, or if prospecting or mining extends into habitat areas, species will need to be relocated, and relevant permits required.

3.1.5 Forest Act (No. 12 of 2001)

Unless approved by the Director of Forestry, no vegetation on a sand dune or within 100 m of a watercourse may be removed or damaged without a license. Mining is currently not expected to extend beyond the previous mine footprint on surface, but prospecting activities may. NLZM will therefore need to monitor the areas under prospecting and if areas within 100 m of a watercourse, or sand dunes, are to be impacted, then the relevant licences must be obtained.

3.1.6 Water Act (No. 54 of 1956)

This Act deals with both surface and groundwater, and requires permits for water abstraction and discharge of effluent to be obtained from the Department of Water Affairs. The Act also regulates the quality of water that may be discharged to the environment.

Water was previously pumped out of the flooded old mine workings to allow for prospecting activities, and discharged (assumedly) into the environment. It is unclear what the quality of this water was, the volumes discharged, or the exact location discharged to. There does not appear to have been a permit in place for this discharge.

NLZM has referred to an existing borehole that may be used for the supply of non-potable water for drilling purposes. It is unclear whether a permit is in place for this. It is assumed that the relevant abstraction and discharge permits will be obtained prior to establishment of the mine. No applications have yet been noted.
3.1.7 Atmospheric Pollution Prevention Ordinance (11 of 1976)

This Ordinance deals with air pollution as it relates to general air quality and greenhouse gas emissions, mineral waste, biodiversity, and community health and safety. No scheduled processes may be conducted within Namibia without a registration certificate. This will be relevant for Stage 2 of mining, for the processing of lead and zinc ore, which are scheduled processes. NLZM will need to have in place a registration certificate prior to commencing processing.

3.1.8 National Heritage Act (No. 27 of 2004)

According to this Act, no archaeological remains may be disturbed or destroyed without a permit from the Heritage Council. Archaeological sites have been found within the EPL (as noted in the relevant specialist study), but not within the proposed mine footprint.

3.1.9 Policy for Prospecting and Mining in Protected Areas and National Monuments

The purpose of this policy is to promote the sustainable development of Namibia by guiding prospecting and mining in the country’s Protected Areas and National monuments. It generally permits the granting of EPLs and mining licences in Protected Areas and National Monuments, except areas which are particularly sensitive or are of special ecological or touristic importance. However, each application is considered on a case by case basis.

The Policy further states that a full EIA will usually be required for any prospecting or mining in a Protected Area and/or National Monument. An Environmental Clearance Certificate is required before prospecting or mining may commence.
4 Conclusions and Recommendations

4.1 Closure and rehabilitation

The EIA states that no details of proposed closure and rehabilitation were made available to CCA by the project engineers. CCA has proposed some measures for rehabilitation and closure – these include the removal of topsoil (prior to establishment of infrastructure) and reapplication after removal of infrastructure, fencing of shafts, ripping and de-compacting of ground, stormwater retention dams for the tailings dam, and possibly cladding of tailings dam. A detailed mine closure and rehabilitation plan is still to be developed.

The EIA recommends the setting up of a rehabilitation fund for mine closure. It is recommended that in the absence of Namibian legislation to guide the calculation of closure costing, the South African MPRDA guidelines be used as an indication.

Aspects of mine closure that still need to be addressed include the following:

- Engineering designs must be completed – e.g. stormwater management facilities. It is likely that construction of dirty water collection channels will be complicated by the lack of surface material which can be excavated. Possible solutions may include constructing lined HDPE plastic channels utilising embankments constructed from waste material, combined with rock lined sections excavated through the bedrock. Site specific conditions must be taken into account for the designs;

- The requirements for removal of contaminated infrastructure from the site (specifically the redundant surface infrastructure – it is unclear within the EIA whether this has yet been removed or will be removed at closure);

- Backfilling or surface deposition of waste rock – these options have not been fully investigated, and designs have not been undertaken;

- The contamination potential of waste rock and tailings material (via seepage); and

- Possible options for successful rehabilitation of the tailings dam including rock cladding.

A rough calculation has been undertaken using the MPRDA guidelines (South African), based on conceptual plans and layouts, using 2013 rates. This incorporates the following very basic assumptions:

- Tailings dam footprint includes combination of old and existing/new tailings dams – both sites will need to be rehabilitated

- Six ventilation shafts (surface area 2.1 m x 2.4 m) and one decline shaft (surface area 3.0 m x 3.5 m) will be filled to a depth of 10 m

- Provision made for 12km of access road

- No waste rock dump noted on surface therefore excluded – still to be determined

- Entire project footprint will be fenced

- Contaminated land assessment to be undertaken at closure
The closure cost calculation indicates an approximate costing of South African Rand (ZAR) 14.7 million, excluding VAT – which would be relevant if NRR undertakes closure themselves. Should the government be expected to undertake rehabilitation and closure (or if closure is premature) then the cost would include VAT and contingencies, and would increase to ZAR 20.5 million.

4.2 Tailings material management

The site of an existing tailings dam will be used to deposit the final tailings material. Specialists have recommended lining the tailings dam or grouting any fractures. It is unclear how lining might be undertaken without the removal of existing tailings material.

Rehabilitation and closure of the tailings dam is unknown. Revegation success is unlikely due to harsh climate and chemicals / reagents within the tailings material. Cladding with waste rock has been suggested, to limit visual impacts and possibly prevent wind erosion and deposition of tailings dust further away from the tailings dam. Further investigations into rehabilitation and closure are recommended.

4.3 Acid rock drainage (ARD) and other contamination

Specialists have indicated that there is potential for acid rock drainage (ARD) from the sulphide minerals. Acid will likely be generated but may be neutralised by host rock carbonate minerals. Acid base accounting (ABA), net acid generation (NAG) tests and synthetic precipitation leaching procedure (SPLP) tests have been recommended. These should be undertaken prior to mining.

It is recommended that continued groundwater sampling and monitoring, with additional monitoring wells, be implemented to assess the levels and quality of groundwater.

4.4 Dust deposition and contaminated soil

Dust deposition is a significant concern, because of the metals composition of the tailings dust. Tailings material from the old tailings dam has contaminated soil up to 8 km from the project site, to a level of 35 cm or deeper in some cases. Elements of concern (all exceeding international guidelines) include lead, zinc, cadmium, copper, nickel, arsenic and manganese, and the existing levels of contamination are regarded as hazardous. It is expected that dust from the proposed tailings dam could contaminate areas more than 10 km away.

It is recommended in the air quality specialist report that the tailings dam be watered shortly before easterly wind conditions are expected, and that different dust suppression chemicals be tested to possibly speed up crusting of the tailings dam surfaces. These actions require further investigation. Wind fences should be considered in a north-south direction. Complete rock cladding / geotextile combination on the tailings dam at closure should minimise post-closure dust deposition.

4.5 Rezoning

The listed activity addressing rezoning was included in the EIA. It is however recommended, that unless already undertaken (of which there is no indication), that NLZM obtain guidance and assistance from government and regional planning consultants with respect to the rezoning of the project site from conservation to industrial / mining.
The National Planning Commission (NPC) must be involved in decision making processes regarding land use and zoning. The NPC was not included as an interested and affected party for the project.

4.6 Permits and licences and associated obligations

A number of permits and licences are still required – it is anticipated that these will be applied for prior to their requirement:

- Permits for the transport, storage, and disposal of hazardous materials
- Permits for abstraction or discharge of water
- Registration certificate (air pollution) for the processing plants
- Permits may be required in terms of the Nature Conservation Ordinance or Forests Act should sand dunes or watercourses be affected, or for relocation / destruction of protected plant species

It is noted above that the EPL has been renewed (possibly) for the last time, and that a mining right must be applied for by 17 April 2014. Should this not be possible, NLZM is recommended to obtain clarification in writing that the EPL can be renewed for a further 2 years.

The reviewed Environmental Contract (2001) for the EPL was incomplete, and full obligations thereof are unknown. There is reference to reports that must be submitted to the Ministry of Mines and Energy. This should be followed up by NLZM.

4.7 Biodiversity and heritage

It is recommended that awareness of fauna, flora and heritage for prospecting and mining staff and contractors be raised, and be included in training and documentation. Remaining within the existing footprint of the mine will likely ensure minimal impacts. Impacts on biodiversity should be monitored and reported on. Ongoing monitoring of the prospecting and mining footprint must occur to ensure that impacts are minimised and if sensitive areas are to be affected, management measures must be implemented. Specialists may need to be reappointed to assist.

All retention dams and ponds must be fenced to prevent attracting wildlife, both because the water is likely to be contaminated and to prevent human-wildlife conflicts. Fencing should be suitable to prevent both large and small wildlife from entering.

4.8 Waste management

Solid waste management should consider alternatives to incineration, preferably formalisation and use of licenced facilities in Swakopmund (general waste) and Walvis Bay (hazardous waste).
5 International best practice

This section provides a very high level comment on current compliance with, and recommendations for compliance with, international best practice, including the Equator Principles and International Finance Corporation (IFC) Performance Standards. The World Bank Group Environmental, Health and Safety (EHS) Guidelines have been excluded from this high level review. General information is provided within each section as a guideline for interpretation of the various requirements.

5.1 Equator Principles (2013)

Excerpt from Equator Principles (2013): The Equator Principles apply globally and to all industry sectors. The Equator Principles apply to the four financial products described below when supporting a new Project:

- **Project Finance Advisory Services** where total Project capital costs are US$10 million or more.
- **Project Finance** with total Project capital costs of US$10 million or more.
- **Project-Related Corporate Loans** (including Export Finance in the form of Buyer Credit) where all four of the following criteria are met:
  - The majority of the loan is related to a single Project over which the client has Effective Operational Control (either direct or indirect).
  - The total aggregate loan amount is at least US$100 million.
  - The EPFI’s individual commitment (before syndication or sell down) is at least US$50 million.
  - The loan tenor is at least two years.
- **Bridge Loans** with a tenor of less than two years that are intended to be refinanced by Project Finance or a Project-Related Corporate Loan that is anticipated to meet the relevant criteria described above.

5.1.1 Principle 1: Review and Categorisation

Projects are classified according to social or environmental impacts, into Category A (potential significant impacts), Category B (potential limited impacts) and Category C (minimal or no adverse impacts). Category A and B projects require a thorough Social and Environmental Assessment as per Principle 2 below.

5.1.2 Principle 2: Social and Environmental Assessment

A Social and Environmental Assessment\(^3\) must address the relevant social and environmental risks and impacts of the proposed project. The Assessment Documentation should propose measures to minimise, mitigate and offset adverse impacts. An EIA has been undertaken for the NLZM, and an EMP compiled to address management of impacts. Labour, health and safety risks have not been thoroughly addressed in the EIA, although these are generally out of scope for EIAs in Namibia.

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\(^3\) Environmental and Social Assessment (Assessment) is a process that determines the potential environmental and social risks and impacts (including labour, health, and safety) of a proposed Project in its area of influence.
5.1.3 Principle 3: Applicable Social and Environmental Standards

The Assessment process must address compliance with relevant host country laws, regulations and permits relating to environmental and social issues. Refer to Section 3. For Non-Designated Countries (such as Namibia), the Assessment process must evaluate compliance with the applicable IFC Performance Standards and World Bank Group EHS Guidelines.

5.1.4 Principle 4: Environmental and Social Management System and Equator Principles Action Plan

An Environmental and Social Management System (ESMS)\(^4\) must be developed for the project. An Environmental and Social Management Plan (ESMP)\(^5\) must also be compiled, to address issues raised in the Assessment process. An EMP has been compiled in accordance with Namibian legislation, and in concept meets some of the requirements of an ESMP. The EMP is considered a working document and will be updated as and when necessary.

5.1.5 Principle 5: Stakeholder Engagement

Effective Stakeholder Engagement must be undertaken as an ongoing process in a structured and culturally appropriate manner. Stakeholder engagement was undertaken to meet the requirements of Namibian legislation. Considering the National Park status of the project site, it is recommended that additional engagement be held with the relevant authorities and relevant environmental organisations to ensure that their requirements and recommendations for operation and closure are understood and implemented. A Stakeholder Engagement Plan must be developed to ensure that ongoing (and detailed) stakeholder engagement is planned for and implemented.

5.1.6 Principle 6: Grievance Mechanism

A Grievance Mechanism is required to be developed as part of the ESMS (above). It must be relevant to both the community and employees. This has not yet been compiled.

5.1.7 Principle 7: Independent Review

For Project Finance, where the total project capital cost is USD10 million or more, an independent consultant will carry out an independent review of the Assessment Documentation, including the ESMPs, the ESMS, and the Stakeholder Engagement process documentation, and assess Equator Principles compliance. This review constitutes a Principle 7 review in concept, albeit on a very high level. A more detailed review is required to fully comply with Principle 7. The independent consultant should also propose a suitable Action Plan capable of bringing the project into compliance with the Equator Principles.

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\(^4\) Environmental and Social Management System (ESMS) is the overarching environmental, social, health and safety management system, applicable at Corporate or Project level. The system is designed to identify, assess and manage risks and impacts in respect to the Project on an ongoing basis. The system consists of manuals and related source documents, including policies, management programmes and plans, procedures, performance indicators, and responsibilities with respect to the environmental or social issues, including Stakeholder Engagement and grievance mechanisms. It is the overriding framework by which an ESMP and/or Equator Principles AP is implemented.

\(^5\) Environmental and Social Management Plan (ESMP) summarises the client’s commitments to address and mitigate risks and impacts identified as part of the Assessment, through avoidance, minimisation, and compensation / offset. This may range from a brief description of routine mitigation measures to a series of more comprehensive management plans (e.g. water, waste, resettlement, emergency response, decommissioning). The ESMP definition and characteristics are broadly similar to those of the “Management Programmes” referred to in IFC Performance Standard 1.
5.1.8 Principle 8: Covenants
Covenants will form part of the commitments and actions required by the Lender/s.

5.1.9 Principle 9: Independent Monitoring and Reporting
The Lender/s will require that an independent consultant is appointed to verify monitoring information after Financial Close and over the life of the loan.

5.1.10 Principle 10: Reporting and Transparency
The Client must ensure that a summary of the ESIA is accessible and available online, and must publicly report GHG emission levels during the operational phase of the project, if emitting over 100,000 tonnes of CO₂ equivalent annually. This will form part of the commitments required by the Lender/s.

5.2 International Finance Corporation (IFC) Performance Standards (2012)
The Performance Standards provide guidance on how to identify risks and impacts, and are designed to help avoid, mitigate, and manage risks and impacts as a way of doing business in a sustainable way, including stakeholder engagement and disclosure obligations.

In addition to meeting the requirements under the Performance Standards, “clients” (Borrowers) must comply with applicable national law, including those laws implementing host country obligations under international law.

5.2.1 PS 1: Assessment and Management of Environmental and Social Risks and Impacts
PS 1 establishes the importance of integrated social and environmental assessment, effective community engagement, and management of social and environmental performance throughout the life of the project. A process of environmental and social assessment must be conducted, and an ESMS (see above, footnote 4), appropriate to the nature and scale of the project and risks, must be established and maintained. The ESMS will incorporate the following elements: (i) policy; (ii) identification of risks and impacts; (iii) management programmes; (iv) organisational capacity and competency; (v) emergency preparedness and response; (vi) stakeholder engagement; and (vii) monitoring and review.

The site is a brownfields site, and the baseline condition is considered impacted. An EIA has been undertaken to recommission the old mine and to extend workings underground. No formal Environmental and Social Management System (ESMS) has been developed for the project. Elements of the ESMS have been undertaken, including items identification of certain risks and impacts, development of certain management programmes, reporting on stakeholder engagement to date, monitoring and review, and the development of an EMP. Items still to be addressed include: policy, formalised management programmes, organisational capacity and competency, stakeholder engagement plan (ongoing, future), formalised monitoring and reporting, and emergency preparedness and response.
5.2.2 PS 2: Labour and Working Conditions

PS 2 highlights the importance of worker-management relationships; non-discrimination and equal opportunity; compliance with national labour and employment legislation; prohibiting child and forced labour; and promoting safe and healthy working conditions. These aspects have not been addressed within the EIA (out of scope). There is no indication whether relevant policies are in place to ensure that PS 2 is complied with.

The EIA recommends that an occupational health and safety specialist should be consulted with or employed, to ensure compliance with relevant national legislation, specifically relating to exposure of employees to hazardous substances such as lead and silica. Further investigations and management are required considering the existing contamination level being classified as hazardous.

5.2.3 PS 3: Resource Efficiency and Pollution Prevention

PS 3’s objectives are to avoid or minimise adverse impacts on human health and the environment by avoiding or minimising pollution from project activities, and to promote the reduction of emissions that contribute to climate change. Technically feasible and cost effective measures must be implemented to improve efficiency in consumption of water and energy. Infrastructure, design or technology alternatives that may reduce greenhouse gas emissions must be considered.

Little information has been made available regarding infrastructure specifications. The use of diesel generators to provide electricity for the plant will generate significant carbon emissions. Alternative sources of energy should be considered and investigated. Greenhouse gas emissions should be estimated / quantified.

Release of pollutants to air, water and land must be avoided. The tailings dam is a significant source of pollutants – to both air and land. Adequate measures must be taken to prevent the spread of dust deposition from the tailings dam. A hazardous materials management plan should be compiled, addressing production, transportation, handling, storage and use. All contaminated water must be contained on site, and treated if necessary (to comply with national water requirements) prior to discharge.

Considering the historical contamination of the site, NLZM must determine the responsibility for mitigation of this contamination, and the expectations regarding mitigation and clean up. If NLZM is deemed to be legally responsible, then liabilities must be dealt with in accordance with national legislation; where liabilities are unclear, collaboration/coordination with the Namibian government must be sought. Annual soil monitoring has been recommended in the EIA/EMP, and this will assist with determining the level of contamination requiring clean-up. It is recommended that a final contamination study be undertaken at closure.

The EMP states that waste might be incinerated on site. It is recommended that alternatives are considered, and that waste instead be transported to licenced facilities in Swakopmund (general waste) and Walvis Bay (hazardous waste). Temporary storage facilities must be adequately designed and constructed to ensure no contamination or pests (bunded, covered areas). Contracts allowing this disposal must be formalised.

5.2.4 PS 4: Community Health, Safety and Security

The objectives of PS 4 are to avoid or minimise the risks to and impacts on the health and safety of the local community; and to ensure that the safeguarding of personnel and property is carried out in a legitimate manner that avoids or minimises risks to the community’s safety and security.
No communities live close enough to the site to be directly affected by any contamination or structural safety issues. However, the Dorob National Park is a tourist area, and several tourist companies have in the past driven through the abandoned mine site. The safety of tourists and tourist companies must therefore be addressed. Tailings dams and other mine infrastructure must be made safe and stabilised, during operation and upon closure and rehabilitation. Access must be prevented, for safety and security reasons. Health concerns may arise as a result of deposition of heavy metals from the tailings dam, far away from the project site. Serious efforts must be made to contain all contamination to the existing footprint. Other potential safety and security hazards to the community must be considered and addressed.

5.2.5 PS 5: Land Acquisition and Involuntary Resettlement

Not relevant for this project as no resettlement will be occurring.

5.2.6 PS 6: Biodiversity Conservation and Sustainable Management of Living Natural Resources

The objectives of PS 6 are to protect and conserve biodiversity, and to promote sustainable management and use of natural resources. While the site is considered heavily contaminated as a result of poor historical mining practices, it is now located within a (recently proclaimed) National Park. This is a protected area in terms of Namibian legislation, although the conditions attached to this status are vague and not yet formalised in legislation. Written permission must be obtained from the Minister of Mines and Energy and the Directorate of Parks and Wildlife.

The surrounding areas can be considered largely natural (as per PS 6), but the footprint itself has been severely modified, to the extent that the biodiversity value prior to mining is unknown. NLZM must ensure that all impacts are limited to the existing footprint, and that awareness is raised of sensitive fauna and flora in the area. Particularly sensitive sites must be designated as no-go areas, and must be avoided by prospecting teams as well.

The impact of contamination on fauna and flora, particularly from dust deposition off the current footprint, must be assessed. The rate of recovery in desert ecosystems is extremely slow, and impacts should be prevented and avoided as far as possible.

PS 6 requires that projects being undertaken within legally protected areas must implement additional programmes to promote and enhance the conservation aims and effective management of the area. This must still be investigated for the NLZM.

5.2.7 PS 7: Indigenous Peoples

No Indigenous Peoples are considered to be living within the project area.

5.2.8 PS 8: Cultural heritage

The objective of PS 8 is to protect cultural heritage from adverse impacts of project activities, and to support its preservation.

The specialist identified several archaeological sites within the EPL, but none within the footprint area. As with biodiversity, awareness must be raised and training implemented as to the sensitivity of these areas, the procedures to follow in case additional sites are discovered. These areas must be designated as no-go areas and must be avoided by prospecting teams as well.