NI 43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project

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1 Summary

1.1 Introduction

The Karowe diamond Mine is an existing open pit diamond mining operation that commenced operations in April 2012 with a projected open pit mine life to 2026.

The Karowe Diamond Mine is 100% owned by Lucara Diamond Corporation (Lucara) through its 100% owned subsidiary Boteti Mining (Pty) Ltd (Boteti).

Lucara commissioned Royal HaskoningDHV to complete a Preliminary Economic Assessment (PEA) for the Karowe Underground Project. The purpose of this study was to undertake preliminary mine designs and costing of an underground extension to the existing open pit mine, culminating in an economic evaluation.

This report was prepared in order to fulfil the reporting requirements as stipulated by Canadian National Instrument 43-101 (NI 43-101) Technical Report.

It must be noted that this PEA is preliminary in nature and includes the use of Inferred Mineral Resources which is considered too low confidence to be able to categorize the mining target as Mineral Reserves. As such, the outcome of the PEA is indicative only. This PEA assumes that the current mining licence will be successfully extended to cater for the period of the underground project.

1.2 Property Location and Description

All mineral rights in the Republic of Botswana are held by the State. Commercial mining takes place under Mining Licences issued on the authority of the Minister of Minerals, Energy and Water Resources.

The property is covered by the Mining Licence (ML) 2008/6L issued in terms of the Mines and Minerals Act 1999, Part VI, and covers 1,523.0634 ha in the Central District of Botswana (Figure 1.1). The ML is in north central Botswana, 25 km south of the Orapa diamond mine and 23 km west of the Letlhakane diamond mine, centred on approximately 25° 28' 13" E / 21° 30' 35" S.

ML2008/6L is 100% held by Boteti, a company incorporated in Botswana. The ML was originally issued on 28 October 2008, and was updated on 9 May 2011 to increase the area to the current extent. It is valid for 15 years and gives the right to mine for diamonds. The Government of Botswana holds no equity in the project.

The property lies on the northern fringe of the Kalahari Desert at an elevation of 1,022 m above sea level in central Botswana and is covered by sand savannah which supports a natural vegetation of trees, shrubs and grasses. The land slopes very gently to the north into the Makgadigadi Depression. The dry valley of the now fossil Letlhakane River, directed into the Depression, passes some 18 km to the northeast of the property and is the only notable physiographic feature in the immediate area.

The area around the property is communal agricultural land used mainly for cattle grazing with limited arable farming. Surface rights have been secured over the Mining Licence area and provide sufficient space for rock dumps, tailings dams and mine infrastructure.

Electrical power is supplied to the Karowe Mine through the Botswana Power Corporation’s (BPC) national grid on commercial terms. Water for the mine is derived from a strong aquifer at the contact of the Ntane Sandstone Formation and the overlying Karoo basalt.

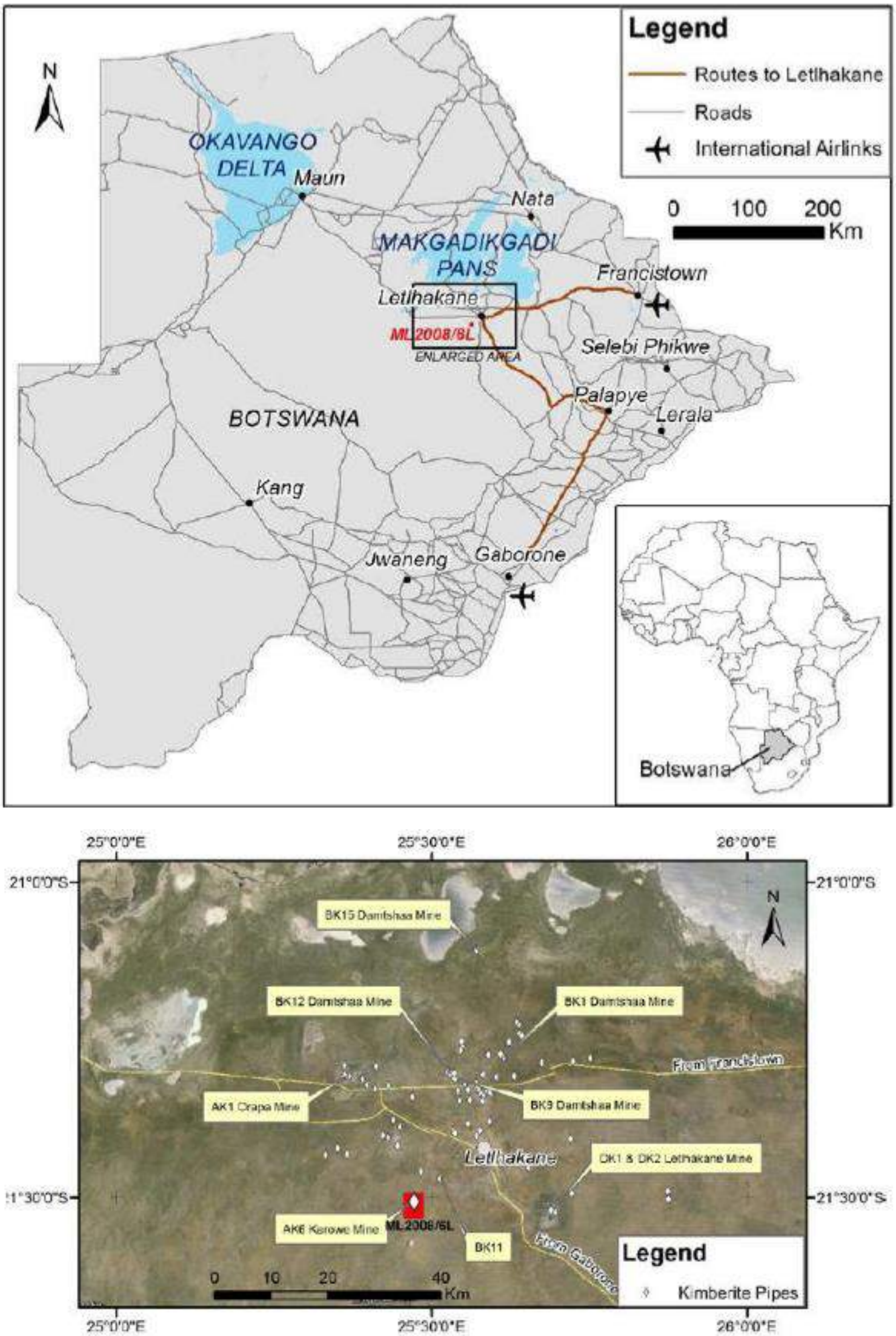

Figure 1.1 – Locality Map of the Karowe Diamond Mine

1.3 Geology

The Karowe Mine is based on the AK6 kimberlite pipe, which is part of the Orapa Kimberlite Field ("OKF") in Botswana. The bedrock of the region is covered by a thin veneer of wind-blown Kalahari sand and exposure is very poor. Rocks close to surface are often extensively calcretised and silcretised due to prolonged exposure on a late Tertiary erosion surface (the African Surface) which approximates to the present-day land surface.

The OKF lies on the northern edge of the Central Kalahari Karoo Basin along which the Karoo succession dips very gently to the SSW and off-laps against the Precambrian rocks that occur at shallow depth within the Makgadikgadi Depression.

The OKF includes at least 83 kimberlite bodies, varying in size from insignificant dykes to the 110 ha AK1 kimberlite which is Debswana's Orapa Mine. All kimberlite intrusions are of post-Karoo age. Of the 83 known kimberlite bodies, five (AK1, BK9, DK1, DK2 and AK6 which is the Karowe Mine) have been or are currently being mined, and a further four (BK1, BK11, BK12 and BK15) are recognized as potentially economic deposits.

The country rock at the Karowe Mine is sub-outcropping flood basalt of the Stormberg Lava Group (approximately 130 m thick on the Karowe property) which is underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup (approximately 245 m thick on the Karowe property). The Karoo sequence overlies granitic basement.

AK6 is a roughly north-south elongate kimberlite body with a near surface expression of ~3.3 ha and a maximum area of approximately 7 ha at ~120 m below surface. The body comprises three geologically distinct, coalescing pipes that taper with depth into discrete roots. These pipes are referred to as the North Lobe, Centre Lobe, and South Lobe.

The AK6 kimberlite is an opaque-mineral-rich monticellite kimberlite that is texturally classified primarily as fragmental volcaniclastic kimberlite with lesser macrocrystic hypabyssal facies kimberlite of the Group 1 variety. The nature of the kimberlite differs between each lobe, with distinctions apparent in the textural characteristics, relative proportion of internal country-rock dilution, and degree or extent of weathering. The South Lobe is considered to be distinctly different from the North and Centre Lobes which are similar to each other in terms of their geological characteristics. The North and Centre Lobes exhibit internal textural complexity (reflected in apparent variations in degree of fragmentation and proportions of country-rock xenoliths) whereas the bulk of the South Lobe is more massive and internally homogeneous.

The upper parts of all three lobes contain severely calcretised and silcretised rock. This zone is typically approximately 10 m in thickness, but can be up to 20 m in places. Beneath the calcrete and silcrete, the kimberlite is highly weathered. The intensity of weathering decreases with depth with fresh kimberlite generally intersected at about 70 m to 90 m below present day surface.

The geological model and list of geological units are presented in Figure 1.2 and Table 1.1 respectively. A unit within the South Lobe (a variety of M/PK(S)) has been found to be hard, and to produce a very large DMS concentrate. Plant upgrades have been undertaken at Karowe to be able to effectively process this material.

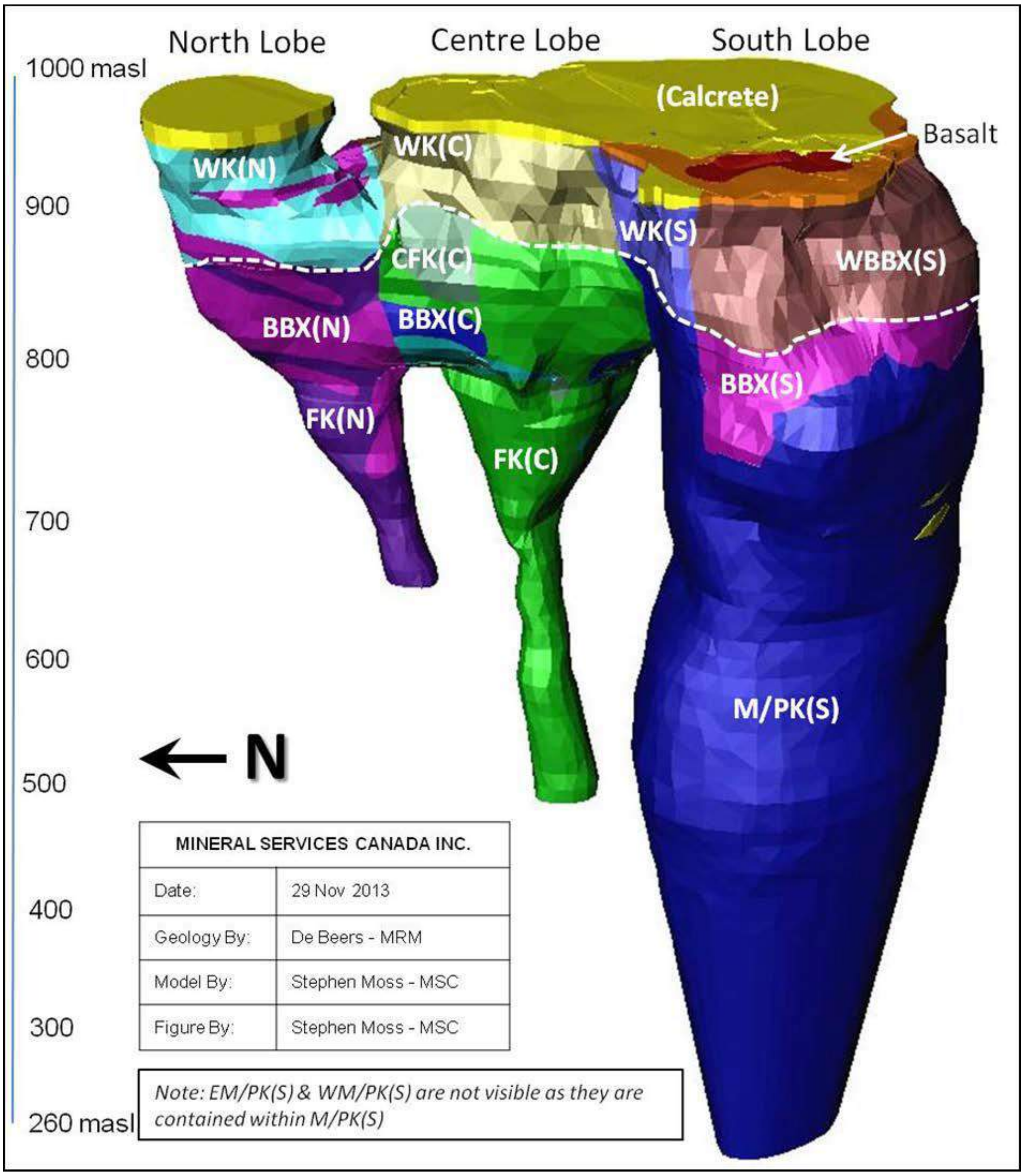

Figure 1.2 – Geological Model of AK6 Kimberlite

Table 1.1 – Kimberlite units identified in AK6 Kimberlite Geological Model

Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country-rock breccia
North	CKIMB	CKIMB(N)	Calcretised kimberlite
North	FK(N)	FK(N)	Fragmental kimberlite
North	KBBX	KBBX(N)	Kimberlite and country-rock breccia
North	WBBX	WBBX(N)	Weathered country-rock breccia
North	WK(N)	WK(N)	Weathered kimberlite
Center	BBX	BBX(C)	Country-rock breccia
Center	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
Center	CKIMB(C)	CKIMB(C)	Calcretised kimberlite
Center	FK(C)	FK(C)	Fragmental kimberlite
Center	KBBX	KBBX(C)	Kimberlite and country-rock breccia
Center	WBBX	WBBX(C)	Weathered country-rock breccia
Center	WK(C)	WK(C)	Weathered kimberlite
South	BBX(S)	BBX(S)	Country-rock breccia
South	CBBX(S)	CBBX(S)	Calcretised country-rock breccia
South	CKIMB(S)	CKIMB(S)	Calcretised kimberlite
South	EM/PK(S)	EM/PK(S)	Eastern magmatic/pyroclastic kimberlite
South	INTBS	INTBS(S)	Large internal block of basalt
South	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite
South	M/PK(S)	17+YIELD	High-yield magmatic/pyroclastic kimberlite
South	WBBX(S)	WBBX(S)	Weathered country-rock breccia
South	WK(S)	WK(S)	Weathered kimberlite
South	WM/PK(S)	WM/PK(S)	Western magmatic/pyroclastic kimberlite

Units occurring in more than one lobe (e.g. BBX, CKIMB) were modelled as separate domains for each lobe. The 17+YIELD material identified in the South Lobe is a variety of M/PK(S) kimberlite defined as a separate domain on account of its very high concentrate yield.

1.4 Mineral Resources

A Mineral Resource estimate was prepared by Mineral Services Canada Ltd (MSC) and published by Lucara on 18th December 2013 (Lynn et al, 2014). No changes have been made to the geological model, bulk density tonnage estimates are based on open pit depletions since 2013 and impact the Indicated resource. The grade estimates made in 2013 were updated by MSC in 2016. The value models of 2013 have been updated in this report based on the substantially larger diamond sales dataset now available following an additional 4 years of active production. These updates to the grade and value estimates have been incorporated by MSC into the 2013 block model in which the original resource estimate was hosted, forming the basis for this PEA. Details of these revisions, and the basis for all estimates, are provided in Section 14. The Mineral Resource statement is presented in Table 1.2.

Table 1.2 – Mineral Resource Statement

Resource	Volume (Mm^3)	Density (tpm^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	$/ct
North Lobe	0.64	2.48	1.58	0.24	15	221
Centre Lobe	1.68	2.57	4.30	0.70	16	400
South Lobe	11.10	2.87	31.82	4.59	14	730
Working SP	1.23	1.87	2.30	0.25	11	437
LOM SP	1.19	1.87	2.23	0.08	4	547
Indicated Total	**15.84**	**2.67**	**42.23**	**5.86**	**14**	**655**
Centre Lobe	0.08	2.58	0.20	0.03	15	400
South Lobe	6.87	2.96	20.37	2.95	14	730
Inferred Total	**6.95**	**2.96**	**20.57**	**2.98**	**14**	**727**

Statement of the estimated remaining Mineral Resource Statement in the AK6 kimberlite deposit as of December, 2016. SP = Stockpile. LOM = Life of Mine. Volume, tonnes and carats are reported in millions (M)

1) *Based on a recoverable grade model (1.25mm bottom cut off size) as revised in 2016*
2) *Diamond price is based on diamonds recoverable with current Karowe plant process and sales value data from 2017 and historic sales*
3) *Effective Date September 15, 2017*
4) *Mineral Resources are reported inclusive of Mineral Reserves*
5) *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability*

1.5 Mineral Reserves

Due to the preliminary nature of the PEA study, this PEA report does not state a Mineral Reserve.

1.6 Geotechnical

The planned open pit to underground mining transition at Karowe Mine has been geotechnically assessed using the information made available at the time of this report including a site inspection of the open pit conditions and existing drill core.

The Karowe project’s geotechnical environment holds both similarities and differences to other operations in the area (Orapa and Letlhakane mines operated by Debswana). On the positive side the following is noted:

- The red mudstone layers located in the Mosolotsane sandstone unit are significantly thinner than those intercepted at other mines.
- Mudstones in both the Thlabala and Tlapana units are more competent and have significantly less swelling clay minerals and higher strengths.
- South lobe kimberlite has high rock strength. This is a positive attribute in terms of development and support requirements during kimberlite extraction, but may pose some risk to natural caveability during mining.

Of general concern, the following is noted:

- Carbonaceous zones within the Tlapana mudstone unit are vertically extensive in most of the drill holes with intervals up to 80 metres in vertical thickness. The carbonaceous zones are recovered intact and appear competent but very quickly degrade.
- The safe development of excavations through the Tlapana mudstone unit where the thick carbonaceous shales persist.

1.7 Hydrogeology

Karowe mine penetrates three major aquifers; The Ntane Sandstone, The Mosolotsane sandstone and mudstones, and the Mea sandstone and weathered Granite contact zone. Total maximum inflow for the open pit and underground mine will be between 3.2 Mm^3/yr and 6.7 Mm^3/yr depending on rates of mining and how much water is extracted up-gradient. There will be minor seepages from the waste rock dumps and slimes dams unless intercepted.

The open pit is currently being dewatered using pit perimeter boreholes and the water level is being held below pit bottom. There are 12 operating dewatering boreholes to 280 mbs soon to be augmented with 4 new design dewatering boreholes. The monitoring network comprises four pit perimeter monitoring boreholes and nine point piezometers.

To obtain a more accurate estimate of total inflows for life of mine the conceptual model requires the addition of hydraulic parameters for the units below the Mosolotsane followed by numerical modelling. An investigation programme is in progress and the updated numerical model will be completed in January 2018.

The probable impact of the mine on regional ground water levels will be up to 5 km radius from the mine lease perimeter (KLMCS 2007). However this will be more accurately predicted using the updated numerical model.

Underground mine dewatering can be achieved using a combination of deeper pit perimeter boreholes and early access tunnels. A ring tunnel will dewater both the underground mine and the open pit to 400 mbs. A combination of vertical raise boreholes and depressurisation core holes will be used to dewater to 720 mbs.

An evaluation (trade–off) of early access tunnel development that can be used to dewater the Karowe open pit and underground mine should be undertaken together in advance of the Karowe underground mine.

Pre-mining dewatering is a prerequisite to successful mining and Karowe has the opportunity to implement a successful dewatering and depressurization schemes ahead of mining.

1.8 Mining

Currently the Karowe AK6 Diamond pipe deposit is being mined by open pit. A recent Whittle run carried out by RHDHV has confirmed that the current economic cut off for open pit mining in comparison to underground mining is at 690 meters above mean sea level (“mamsl”) as was determined previously. The Whittle run employed the understanding of the current mine costs for open pit mining including completing the Cut 2 push back.

In terms of this PEA, the bulk mining method best suited for underground mining, measuring an average 200 m diameter to 420 mamsl (590 meters below surface “mbs”), was either sub level caving (SLC) or block caving (BC). An initial high level cost comparison, encompassing the known geological, hydrogeological, geotechnical and the physical attributes of the deposit, indicated preference for SLC at the PEA stage as the operating cost advantage for BC is offset against a high upfront capital requirement. The difference between the 2 methods will require an in depth trade off exercise at the preliminary feasibility study (PFS) stage when improved data will become available in the geological, hydrogeological and geotechnical fields.

It is understood that the open pit will be able to maintain the current production rate of 2.5 mtpa until 2026. The principal objective for the underground operation will be to ensure economic maintenance of this production rate from 2026 onwards.

Two levels below the pit bottom and an additional four levels of the underground mining will be mined by a derivation of sub level open stoping that will daylight into the pit bottom. This mining method is sub level open stoping (SLOS).

The mine design for both the upper levels SLOS and the following SLC mining once general caving of the host rocks commences, will be similar. The sublevel interval will be 25m, which is the international norm for this method. Owing to the documented hardness and strength of the kimberlite at 156 Mpa it is intended to evaluate increasing the sub level interval to 30m at the PFS stage. This will be subject to maintaining the blast hole ring drilling and charging capabilities.

The primary access to the underground kimberlite will be a boxcut portal at surface housing twin 6.0 m wide x 6.3 m high access declines to cater for large haul 60 tonne trucks loaded by 21 tonne LHD’s and for ventilation intake airways. The twin decline system will be required to facilitate a production rate of some 7 500 tonnes per day of kimberlite and waste and each will traverse variable ground conditions some 2000 m to reach the start of two diametrically opposed spiral production ramp developments.

The mine design was done in Datamine Studio 5D Planner. At this level of study, only the main infrastructure was designed. The main development ends were sequenced in 5D Planner and this information was sent to the EPS Scheduler to use for the Mine Schedule. The design was also evaluated against the Geological Block Model to determine the grade, tonnes and dollar per tonne. The resulting production schedule is shown in Table 1.3 below.

Table 1.3 – Underground Production Summary

Name	Totals	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
South Decline Metres	***4 902***	827	827	611	414	414	414	317	207	207	207	207	207	44	-	-	-	-
North Decline Metres	***4 841***	485	827	782	414	414	414	415	414	310	207	161	-	-	-	-	-	-
Access Crosscut Metres	***2 741***	-	-	176	179	386	381	284	299	313	224	229	133	138	-	-	-	-
Rim Tunnel Metres	***12 391***	-	-	481	887	1 805	2 022	1 866	1 066	911	876	848	825	803	-	-	-	-
Crosscut Metres	***37 965***	-	-	358	1 427	5 151	7 243	7 068	3 394	3 110	3 129	2 640	2 411	2 036	-	-	-	-
RoM Tonnes	***24 726 833***	-	-	37 357	457 104	1 242 109	2 487 983	2 487 589	2 471 729	2 455 653	2 454 909	2 427 720	2 488 695	2 460 606	2 272 565	960 535	22 281	37 357
RoM cpht		-	-	7.67	8.00	8.85	11.22	12.37	13.24	12.26	11.35	12.91	13.13	12.44	12.44	12.41	11.59	7.67

1.9 Processing

The existing process plant has undergone an upgrade in 2015 (the Phase 2 capital project) which was intended to:

- Protect and enhance recovery of large diamonds.
- Enhance comminution performance to maintain design throughput with harder kimberlites.
- Minimise recovery yields when treating these harder kimberlites.

Whilst these process enhancements should cater for the expected increase in kimberlite hardness, with the change from open pit to underground mining and increasing depth. Hardness may increase further putting additional stress on the comminution circuit and recovery yields. Additional information on kimberlite metallurgical characteristics that will affect plant throughput and recoveries needs to be collected and assessed during the ongoing geotechnical drilling campaign. Any additional comminution requirements would lead to an increase in energy requirements.

Further process plant upgrades (the Phase 3 project) have been completed and have been in operation since Q3 2017. These upgrades include an XRT circuit treating +50-125mm material, prior to milling that enables recovery of larger diamonds as early as possible in the process in addition to reducing the risk of diamond damage. A new XRT circuit has also been introduced to treat the 4-8mm fraction, previously sent to DMS, thus reducing the load on the DMS and to cater for higher yield material expected in future. Kimberlite physical properties gathered from the recommended drilling and sampling campaign will provide an indication of whether or not further enhancements will be required to successfully treat potentially higher yields from underground kimberlite.

Once all the process enhancements have been fully optimised, planned maintenance requirements for all the newly commissioned sections should be reviewed to minimise unplanned stoppages. Overall Plant Utilisation (OPU) must be maintained at target levels of 85% at design tonnage throughput, to ensure a suitable basis for the treatment of the future underground resource.

It is recommended that a Geometallurgical data base be established in order to record all relevant data that is gathered from the drilling programmes. In addition, data from current operations should be recorded as the pit goes deeper and factors noted such as kimberlite hardness, density and particle size distribution. This database would then be used to provide the expected underground ROM feed characteristics envelope as inputs for modelling the process. In this way, plant performance can be predicted and enhancements to the flowsheet highlighted which may be required to achieve target throughputs and recoveries. The potential for tramp metal to report with the plant feed must also be assessed once the mining method has been chosen so that additional detection and removal equipment can be included in the plant improvements.

1.10 Capital and Operating Costs

1.10.1 Capital cost estimates

Table 1.4 below shows a summary of the estimated capital required:

- Development capital includes the procurement of necessary equipment and estimated development costs required for a twin decline system, the associated ramps, rim tunnels and access cross cuts.
- Engineering capital includes the estimated costs for additional surface infrastructure to support the underground mine requirements and to integrate the use of existing surface infrastructure. In addition, it includes the electrical supply upgrade, underground pump stations and general project development costs.
- Tailings storage facility estimate accounts for the expansion of the existing facility to ensure capacity for the underground mining requirements.
- Closure costs is the provision for the closure in accordance with all applicable legislation at the end of the underground operations.
- Capital provisions included the annual working (stay in business) capital requirements.

- A 25% contingency was applied to all capital expenditure, particularly to account for the expected difficulty of developing through the host rock. Contingency is excluded in the last year of life of mine.

Table 1.4 - Summary of Capital Costs

Area	Capital (2018-2025) (US$ '000)	Capital (2026-2037) (US$ '000)	Area Totals (US$ '000)
Development Capital	$86 256	$77 416	$163 672
Engineering Capital	$48 530	$14 874	$63 404
Tailings storage facility	$20 920	$6 781	$27 701
Closure Costs	$0	$20 000	$20 000
Capital Provisions	$0	$27 000	$27 000
Net Total Capital	$155 706	$146 071	$301 777
Contingency	$38 926	$31 418	$70 344
Total Capital	**$194 632**	**$177 489**	**$372 121**

1.10.2 Operating cost estimates

The operating cost estimates were based on a combination of experience, reference projects, benchmarked operating costs, budgetary quotes and factoring as appropriate for a preliminary study.

A summary of the life of mine operating costs for the underground mine is shown in Table 1.5. The costs are shown in United Stated Dollars (US$) and a 10% contingency is included.

Table 1.5 – Summary of Operating Costs

Description	Total Operating Costs LOM ($000's)	Average Cost per Tonne ($/t processed)
Mining*	766 532	31.00
Processing	292 271	11.82
Engineering	50 786	2.05
Contingency (10%)	110 959	4.49
Subtotal	**$1 220 548**	**$49.36**
Sustaining Development Capital	119 198	4.82
All-in Sustaining Costs	**$1 339 746**	**$54.18**

1.11 Economic Analysis

The Mineral Resource Estimate (MRE) and underground mining schedule produced for this PEA study including estimated capital and operating costs have been used as the basis for the financial model for the project. The diamond price for the south lobe based on the 2017 average price model is escalated by 2.5% per annum over the life of the mining schedule proposed in the PEA.

It must be noted that this economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and that there is no certainty that the results of the preliminary economic assessment will be realised.

Table 1.6 shows a summary of the resulting NPV at various discount rates.

Table 1.6 - Summary of NPV Sensitivity

NPV (2018 Basis) SENSITIVITY		
DISCOUNT RATE	**US$ Millions**	
5%	**US$**	**451**
8%	**US$**	**318**
11%	**US$**	**226**

Figure 1.3 shows the NPV sensitivity vs. Revenue, Operating Cost, and Capital expenditure for a variation of 10 percent.

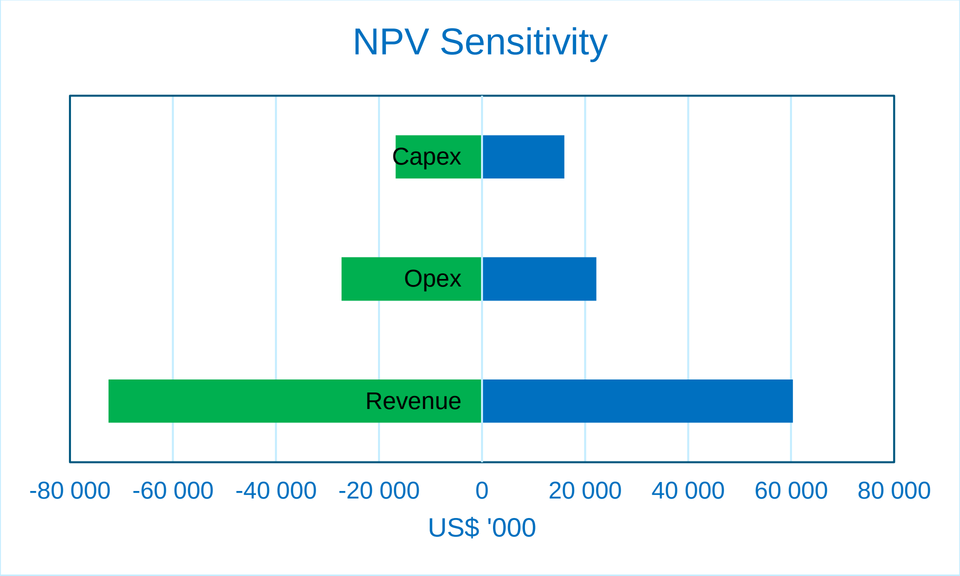

Figure 1.3 – NPV Sensitivity

1.12 Conclusions and Recommendations

The financial model indicates that the underground project has positive economics to its scheduled close in 2037. The estimated NPV is US$451 Million at a 5% discount rate, and a free cash flow of US$820 Million after taxes.

Table 1.7 shows a summary of the underground project economic sensitivities and key operational parameters.

Table 1.7 – Economic Sensitivities and Key Operational Parameters

Parameter	Unit	Base Case
Rough diamond price – South lobe (2017)	US$/carat	$730
Rough Diamond Annual Real Diamond Price Escalator	%	2.5
After-Tax Undiscounted Net Cash Flow	US$M	$820
After-Tax NPV (5%)	US$M	$451
After-Tax NPV (8%)	US$M	$318
After-Tax IRR	%	38.9%
Pre-Tax Undiscounted Net Cash Flow	US$M	$901
Payback Period (pre-tax)	years	2.5

Production	Average Annual
Rough Diamonds million (carats)	2.72
Operating Costs	**US$ per tonne treated**
Kimberlite (US$/t treated)	$49.4
Diamonds (US$/carat recovered)	$407.70
All-In Sustaining Costs[1]	**US$ per tonne treated**
Kimberlite (US$/t treated)	$54.18
Diamonds (US$/carat recovered)	$411.72

Production Costs	$/t
Operating Cost	$49.36
Overhead Costs	$8.93

Karowe mine as an established operation has an existing process plant, engineering infrastructure and an established diamond sales and marketing network. This significantly reduces the risk associated with establishing a new mine. It is RHDHV's opinion that the financial analysis of this PEA of the Karowe underground project has positive economics and warrants consideration for advancement to a Pre-feasibility study (PFS).

2 Introduction

The Karowe Diamond Mine is an existing open pit mine with associated key components including mining related infrastructure, an access road, power and water supply.

2.1 Scope of Work

This Preliminary Economic Assessment Report ("the Report" or the "PEA") has been prepared on behalf of Lucara. Karowe is 100% owned by Lucara through its 100% owned subsidiary Boteti Mining (Pty) Ltd ("Boteti") and commenced open pit operations in April 2012 with a projected life of mine of at least 15 years (9 years remaining life). The opportunity to transition the open pit operation to an underground operation is the subject of this report. The production from Karowe is sold by closed tender both locally and internationally.

The Report complies with disclosure and reporting requirements set forth in the Toronto Stock Exchange (TSX) Corporate Finance Manual, Canadian National Instrument 43- 101 Standards of Disclosure for Mineral Projects, Companion Policy 43-101CP, Form 43-101F1 Technical Report (Form F1) of June 2011 and the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council in November 2010. Unless otherwise stated, all monetary figures expressed in this report are in United States dollars ("US$"), all units are in metric measures, and the coordinate system used is geographic latitude and longitude expressed as decimal degrees with true North bearings. The datum for all maps is WGS84. A glossary of all technical terms and abbreviations is included in Appendix 1.

2.2 Previous Technical Reports

Lynn M D, Nowicki Dr. T, Valenta, M, Robinson B, Gallagher M, Bolton R, Sexton J, (2014) NI 43-101 Independent Technical Report of Karowe Diamond Mine Botswana, 3 February 2014.

McGeorge, I.; Lynn, M.D.; Ferreira, J.J.; Croll, R.C.; Blair, D. and Morton, K. (2010) NI 43-101 Technical Report on the Feasibility Study for the AK6 Kimberlite Project, Botswana. The MSA Group, 31 December 2010

McGeorge, I.; Lynn, M.D.; Ferreira, J.J.; and Croll, R.C. (2010) NI 43-101 Technical Report on the Boteti Kimberlite Project, Botswana. The MSA Group, 25 March 2010

2.3 Qualified Persons

The following serve as the Qualified Persons for this technical report as defined in National Instrument 43-101, Standards of disclosure for Mineral projects.

This report has been compiled by:

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Dr. Tom Nowicki	PhD. P.Geo	Mineral Services Canada
Dr. John Armstrong	PhD. P. Geo	Lucara Diamond Corporation

Dr Tom Nowicki visited the mine on the 3rd and 4th July 2013.

All other Qualified Persons visited the mine from the 9th to 11th September 2017.

Mr Guilluame Johannes Oberholzer is an electrical engineer with 35 years' experience in the engineering industry, 25 Years have been in mining as engineering manager and in the mining consultancy environment as an engineering specialist. He has been involved in numerous feasibility studies (from concept to feasibility). He is registered as a professional engineer with the Engineering Council of South Africa and a senior member of the South African institute of Electrical Engineers. (SAIEE). He has been a Director of Turgis consulting, and later Head of engineering for RHDHV after the RHDHV acquired Turgis.

Mr Norman Blackham has 40 years' mineral's industry experience, mostly with Anglo American and De Beers, covering operations management, projects and discipline leadership in gold, uranium and diamond extraction. Whilst mainly focussed on all aspects of the mineral processing value chain, Norman gained significant experience in mineral resource management and mining operations as well as heading up security and safety audit teams at various operations. As Head of Ore Processing for De Beers Consolidated Mines, a significant part of his role was devoted to development, implementation and auditing of diamond loss prevention and process assurance programmes. He spent three years with Rio Tinto setting up and managing the African regional operations business improvement team which focussed on improving operational efficiencies and cost reduction across the whole production value chain of Rio's operations in Southern Africa. After entering the consultancy field, he has focussed on business improvement and auditing having recently carried out diamond process and security audits in various Southern African operations.

Mr John Anthony Cox has 50 years of experience in the mining industry. The bulk of this experience has been in large underground mines employing a wide range of mining methods suited to mechanisation. A high proportion of this experience has been with large open stoping and caving methods, particularly sub level caving. This experience has extended to depths of 2,500 m at Target and South Deep mines. As General Manager Strategic Planning for Ashanti

Goldfields he was responsible for the JORC compliant resource to reserve generation in 1996 and 1997. He has spent the last seventeen years as a mining consultant for RHDHV in the field of mechanisation and deep level massive mining.

Mr Jody Thompson is a mining engineer specialising in the field of mining rock mechanics and geotechnical engineering. He has seventeen year’s rock mechanics and geotechnical experience mostly in underground hard rock mining but with good exposure to other mining environments including opencast and massive mining operations. Mr Thompson has worked as an operational rock engineer in the gold, platinum and chrome industries where the highest position held was that of Rock Engineering Manager. He has also worked with several major consulting houses in South Africa, where he has served on site specific problems, feasibility studies as well as research and development initiatives. In a more recent role, Jody served the role of Manager to the Worley Parsons RSA rock engineering and geotechnical departments. As an active member of SANIRE, Jody is involved in the development of Rock Engineering in South Africa on a national level and is also an active mentor to junior engineers.

Dr Kym Morton has 36 years’ experience in mining Hydrology, specifically mine dewatering design and risk reduction. She is a Consulting Mining hydrologist with KLM Consulting Services in South Africa with offices in Botswana and the UK. Her qualifications include BSc Honours Kings College, MSc Hydrogeology University College, PhD Imperial College and MBA Imperial College. Her PhD was on the hydrogeology and dewatering design for Finsch Kimberlite Mine and she has worked on all of the De Beers and Debswana mines internationally. She is a registered Chartered Geologist (UK) and Professional Natural Scientist (Pr Sci Nat) in South Africa. Dr Morton did the original dewatering design and water supply for neighbouring mines Orapa, Letlhakane and Damtshaa. Dr Morton has the appropriate relevant qualifications, experience, competence and independence to act as a “Qualified Person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects). Dr Morton undertook the site visit to Karowe in 2006, 2009, 2011 and 2017.

Dr Markus Reichardt holds a PhD in restoration ecology and has 25 years’ experience in the mining industry. For ten years he worked in various corporate and operational roles within the Anglo American Group, with the last role being Corporate Environmental Manager for AngloGold Ltd. He has extensive experience as an ESG analyst working for financial sector clients in Africa and Europe. He has also worked extensively for several major global and South African based consulting houses performing due diligence, feasibility studies, audit and rehabilitation commissions on targets on four continents.

Dr Nowicki has 24 years’ experience as a geoscientist in mineral exploration, evaluation and mining, focussed primarily on exploration for and evaluation of primary diamond deposits. He is a Technical Director and Senior Principal Geoscientist with Mineral Services Canada Inc. and is a registered professional geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC). Dr Nowicki has the appropriate relevant qualifications, experience, competence and independence to act as a “Qualified Person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects); his certificate as a Qualified Person is attached in Appendix 2.

Dr John Armstrong has over 28 years of combined experience in mineral exploration, mining and government and is a registered professional geoscientist with the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories (NAPEGG). Dr. Armstrong has strong capabilities in the assessment and analysis of diamond size distributions, content modelling, and value distributions. He has been an employee of Lucara Diamond Corp since September 2013 and implemented predictive size distribution models for the Karowe Mine supporting the presence and recovery of large diamonds. Dr. Armstrong has the appropriate relevant qualifications, experience, competence and independence to act as a “Qualified Person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects); his certificate as a Qualified Person is attached in Appendix 2, Dr Armstrong is not independent of Lucara Diamond Corporation.

The authors of this report, unless otherwise stated, do not have and do not intend to obtain, any material interest in Lucara or the mineral properties in which Lucara has an interest. Our relationship with Lucara is solely one of professional association between client and independent consultants, unless otherwise stated. This report is prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.4 Principal Sources of Information

The Qualified Persons based their review on information provided by Lucara, along with technical reports by previously engaged consulting firms and other relevant published and unpublished data. A listing of the principal sources of information is included in Section 27 at the end of this Report.

The Qualified Persons have endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the Report is based. A final draft of the Report was also provided to Lucara, along with a written request to identify any material errors or omissions prior to lodgement.

The Report has been prepared on information available up to and including October 2017. The Qualified Persons have provided consent for the inclusion of this Independent Technical Report in public disclosure documents.

3 Reliance on Other Experts

The authors have not independently verified, nor are they qualified to verify the legal status of the Mining Licence that forms the subject of this report and are reliant on the information provided by Lucara. The present status of the Mining Licence is based on copies of the licence documents provided by Lucara. This Report has been prepared on the assumption that Lucara is the lawful holder of the Mining Licence.

Advice was sought from other specialists including:

- Mr. Keith Wilson, BSc Mech Eng, Government Certificate of Competency Mines and Works for general engineering.
- Dr. Alan Guest, Pr.Sci.Nat, PhD (Mining), MSc (Mining), BSc (Mining) for geotechnical studies.
- Mr. Francois Graaf, Chamber of Mines Ventilation Certificate for ventilation studies.
- Mr. Herman Venter, Pr. Tech. Eng, Tailings storage facilities studies.
- Mr. Francois Le Roux, B.Eng (Mining), Mining capital and operating costs.

4 Property Description and Location

4.1 Overview of Botswana

The Republic of Botswana gained independence from Great Britain in 1966 and has subsequently been governed by the Botswana Democratic Party in a multi-party democracy. It has the highest sovereign credit rating in Africa and is one of the world's fastest growing economies.

Botswana is the world's largest diamond producer by value, driven mainly by the large Jwaneng and Orapa Mines owned by Debswana. Mining is governed by the Mines and Mineral Act 17 that came into effect on 1st December 1999 and is considered one of the most competitive and best administered mining legislation in Africa. The mining laws are geared to ensure stability, deregulation and government transparency. Botswana is rated by the *Fraser Institute* (2012) as the best destination in Africa for mining investment and by *Transparency International* as the least corrupt country in Africa.

4.1.1 Types of mineral licence in Botswana

In Botswana, mineral rights are vested in the state. There are four types of mineral licences:

- Prospecting Licence: A prospecting license is valid for an initial period of up to 3 years with 2 renewals each not exceeding 2 years each. At the end of each period the prospecting area is reduced by half or at lower proportion as the Minister may decree. The applicant must have access to or have adequate financial resources, technical competence and experience to carry out an effective exploration programme
- Retention Licence: This licence provides for prospectors who deem a project economically unviable in the short-term. The first three-year licence remains exclusive while a second three-year licence provides limited rights for third parties to reassess a prospect.
- Mining Licence: This licence is initially valid for a period of up to 25 years, as is reasonably required to carry out the mining programme. The holder of a licence may apply for unlimited reviews for a period up to 25 years. Additionally, mineral rights holders may be required to permit the government to hold up to a 15% minority interest in mining undertakings. This will be on commercial terms with the Botswana Government paying its pro rata share of costs incurred.
- Minerals Permits: This permit allows companies to conduct small-scale mining operations for any mineral other than diamonds over an area not exceeding a half square kilometre. It is initially issued for five years, with unlimited renewal periods of up to five years each.

4.1.2 Fiscal regime of Botswana

- The royalty rate on precious stones is 10%.
- There is a negotiated rate of income tax for diamond projects (Section 4.3.2).
- 100% depreciation of capital expenditures is allowed.
- There is a 15% dividend withholding tax on distribution to shareholders.
- Mining equipment and spares are zero-rated, otherwise duties are payable.
- There is 10% Value Added Tax (VAT) which applies to all but zero rated items, and applies to mineral exports.
- There is 15% taxation on revenues for downstream cutting and polishing of diamonds.

4.2 Issuer's Title, Location and Demarcation of Mining Licence

The property is Mining Licence (ML) 2008/6L issued in terms of the Mines and Minerals Act 1999, Part VI, and covering 1,523.0634 ha in the Central District of Botswana. The licence is located in north-central Botswana, 25 km south of the

Orapa diamond mine and 23 km west of the Letlhakane diamond mine. It is centred on approximately 25° 28' 13" E / 21° 30' 35" S.

All mineral rights in Botswana are held by the State. Commercial mining takes place under Mining Licences issued on the authority of the Minister of Minerals, Energy and Water Resources.

ML2008/6L is 100% held by Boteti, a company incorporated in Botswana. The ML was originally issued on 28th October 2008, and was updated on 9th May 2011 to increase the area to the current extent. It is valid for 15 years and gives the right to mine for diamonds. The Government of Botswana holds no equity in the project. The corner points and geographic location is shown in Table 4.1, Figure 4.1 and Figure 4.2.

Table 4.1 – List of Corner Points of ML 2008/6L

Corner Points	Longitude (East)			Latitude (South)		
	Degrees	Minutes	Seconds	Degrees	Minutes	Seconds
A	25	27	17.3	21	29	31.1
B	25	29	13.7	21	29	31.1
C	25	29	13.7	21	31	59.1
D	25	27	17.3	21	31	59.1

WGS84 Datum

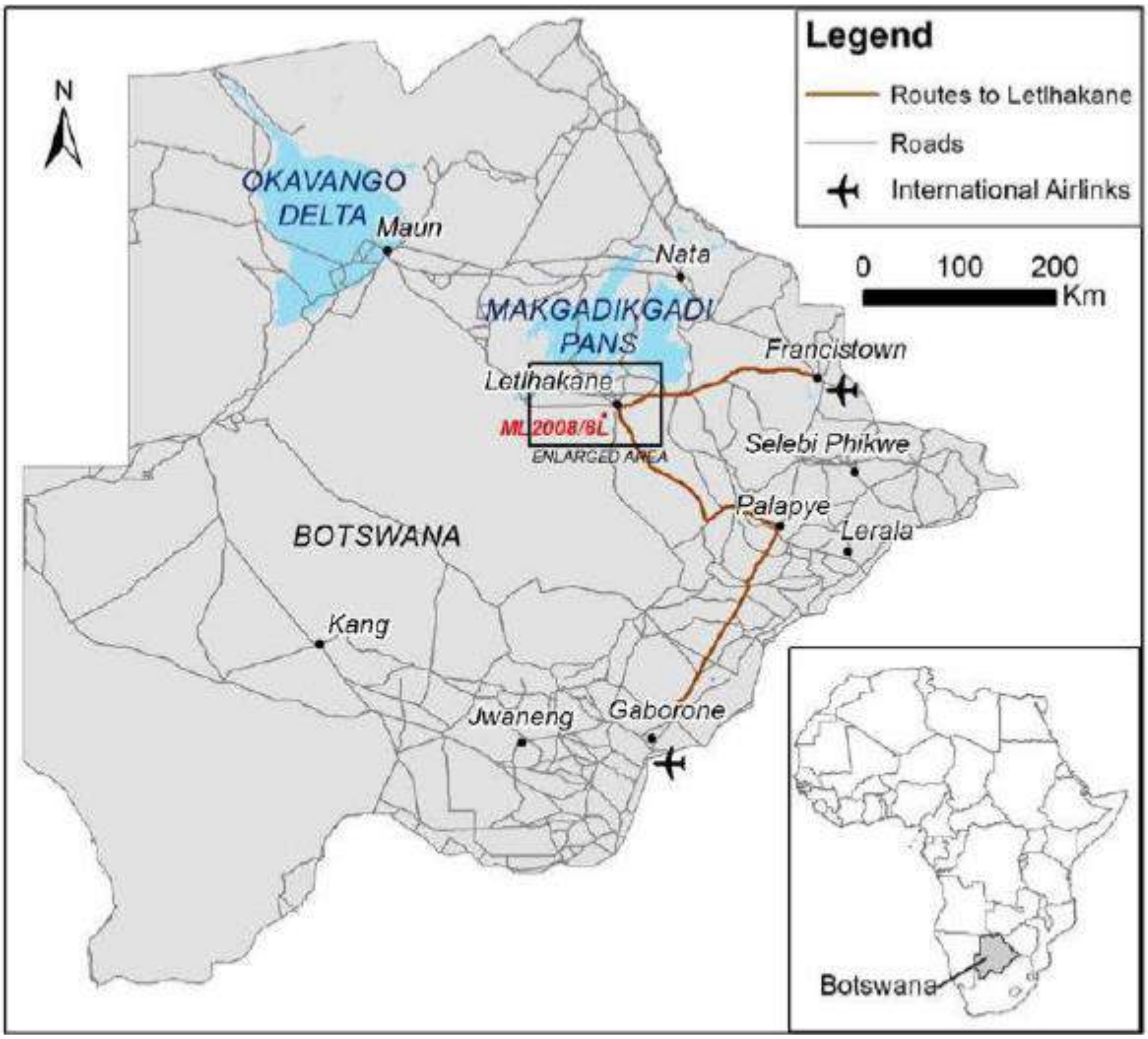


Figure 4.1 – Locality Map

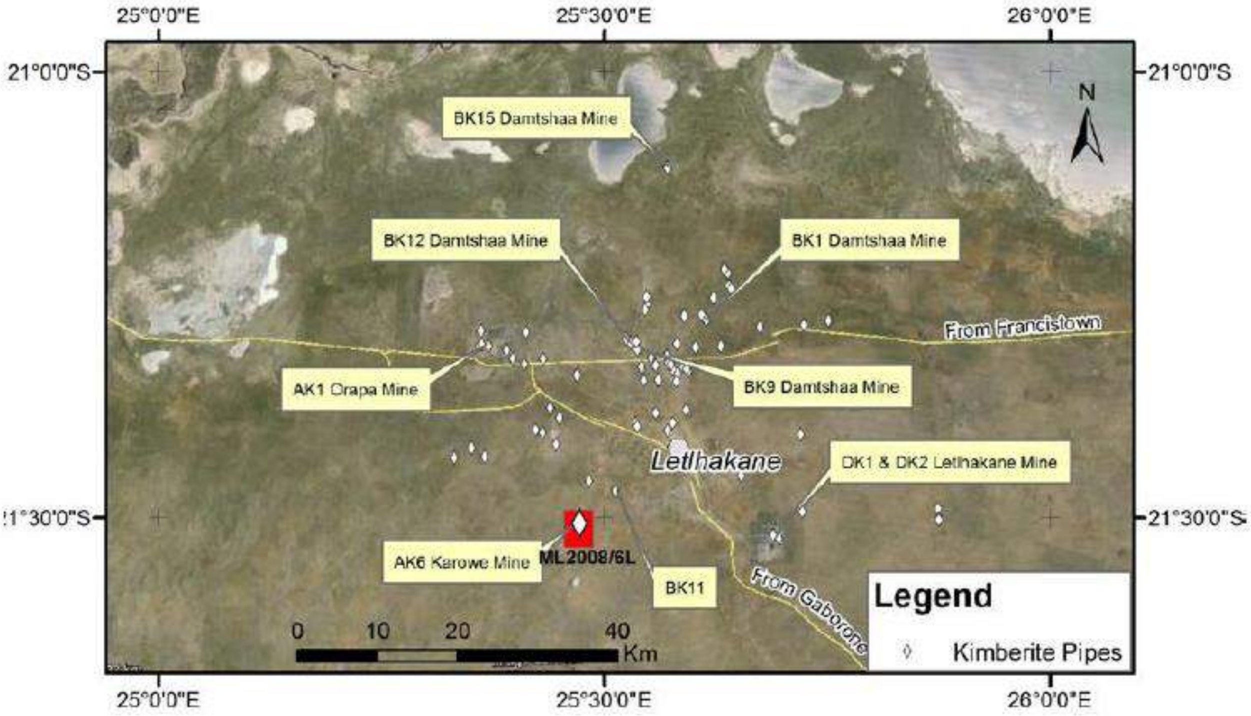


Figure 4.2 – Karowe Mine and other diamond mines in the vicinity

Figure 4.3 is an aerial photograph of the Karowe mine and marked up to highlight the open pit, the stockpiles, waste dumps, tailings dam and tailings dumps. The process plant is located to the east of the open pit.

Figure 4.3 – Aerial photograph of the Karowe Mine

4.3 Permitting Rights and Agreements Relating to Karowe Mine

4.3.1 Surface rights

The surface area of ML2008/6L was originally communal agricultural land administered by the Letlhakane Sub-Land Board, which falls under the Ngwato Land Board, Serowe. It was used for grazing livestock and limited arable farming. Boteti has obtained common law land rights for the ML2008/6L surface area and the access road. These rights will remain in force until 2023.

4.3.2 Taxes and royalties

The Karowe Mine is taxed according to a prescribed schedule of the Income Tax Act. Profits from the Karowe Mine are taxed according to the annual tax rate formula as follows:

> 70-(1500/x) where x is the profitability ratio given by taxable income as a percentage of gross income (provided that the tax rate will not be less than the company rate). Boteti is authorised to offset withholding taxes against the variable Income Tax liability.

A royalty of 10% on actual sales of diamonds is levied by the Government of Botswana.

4.3.3 Obligations

Subject to the provisions of the Mines and Minerals Act, the holder of a mining licence shall:

- Commence production on or before the date referred to in the programme of mining operations as the date by which he intends to work for profit.
- Develop and mine the mineral covered by his mining licence in accordance with the programme of mining operations as adjusted from time to time in accordance with good mining and environmental practice.
- Demarcate the mining area.
- Keep and maintain an address in Botswana.
- Maintain complete and accurate technical records of operations in the mining area.
- Maintain accurate and systematic financial records of operations in the mining area.
- Permit an authorized officer to inspect the books and records of the mine.
- Submit reports, records and other information as the Ministry may reasonably require.
- Furnish the Ministry with a copy of the annual audited financial statements within six months of the end of each financial year.

Boteti has met all of these obligations.

4.3.4 Environmental liabilities

Current environmental liabilities comprise those to be expected of an active mining operation. These include the open pit, processing plant, infrastructure buildings, a tailings dam, and waste rock storage facilities. The environmental permitting and closure plan is discussed in more detail in Section 20.

4.3.5 Permits

A list of permits held or in the process of being acquired by the Karowe Diamond Mine is presented in Table 4.2.

Table 4.2 – Karowe diamond Mine Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIA Permit	DEA/BOD/CEN/EXT/MNE 015(7)	EIA valid. EMP updated in June 2016 and will be reviewed to include phase 3 in 2018	Dept. of Environmental Affairs	EIA Act
Water Rights	B6615, B6622, B5386, B 5387, B5388, B5389, B7933B7934, B7935, B7936, B7937, B7937, B7938, B7940, B7941, B7942	Valid for the duration of the mining licence	Dept. of Water Affairs	Water Act
Waste Carriers License	CRLIC/450/02-1881/17	28/02/2018	Dept. of Waste Management and Pollution Control	Waste Management Act
	CRLIC/450/06-1881/17	30/06/2018	Dept. of Waste Management and Pollution Control	Waste Management Act
	CRLIC/01/04-063/17- SKIP HIRE	30/04/2018	Dept. of Waste Management and Pollution Control	Waste Management Act
	CRLIC/01/011-063/16 - SKIP HIRE	30/11/2017	Dept. of Waste Management and Pollution Control	Waste Management Act
	CRLIC/01/03-063/17 – SKIP HIRE	31/03/2018	Dept. of Waste Management and Pollution Control	Waste Management Act
Incinerator Permit	Application in Progress	Awaiting department of waste management and pollution control to register and licensing the incinerator	Dept. of Waste Management and Pollution Control	Waste Management Act
Borehole Certificates	In Place	Valid for the duration of the mining licence	Dept. of Water Affairs	Boreholes Act
Dumps Classification	All classified	All dumps active	Dept. of Mines	Mines, Quarries, Works and Machinery Act
Surface Rights	LT/SLB/B/1 IV (231)	09/10/2023	Ngwato Land Board	Tribal Land Act
Radiation License	BW0315/2015	Renewed and awaiting certificates	Radiation Inspectorate	Radiation Protection Act

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Waste Facilities & Sewage Plant	Application in Progress	Awaiting department of waste management and pollution control to register and licensing the incinerator	Dept. of Waste Management and Pollution Control	Waste Management Act
License to manufacture explosives	In Place	31/12/2017	Dept. of Mines	Explosives Act
Permit to carry bulk explosives	F35/13, F34/13 and F36/13	31/12/2017	Dept. of Mines	Explosives Act
Magazine License	386:00002948A and 385:00002947A	31/12/2017	Dept. of Mines	Explosives Act
Blasting License for magazine master	In Place	Valid and appointment renewed yearly	Dept. of Mines	Explosives Act

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The area lies on the northern fringe of the Kalahari Desert of central Botswana and is covered by sand savannah which supports a natural vegetation of trees, shrubs and grasses. The trees and shrubs are dominantly mopane (Colophospermum mopane) and tend to form thickets with intervening grassy patches. The natural vegetation has been modified by many years of cattle grazing and limited arable farming.

The property is at an elevation of 1,022 m above sea level and slopes very gently to the north into the Makgadigadi Depression. The dry valley of the now fossil Letlhakane River, directed into the Depression, passes some 18 km to the northeast of the property and is the only notable physiographic feature in the immediate area.

The area around the property is communal agricultural land used mainly for cattle grazing with limited arable farming. Surface rights have been secured over the Mining Licence and provide sufficient space for rock dumps, tailings dams and mine infrastructure.

5.2 Access

The property is accessed by 15 km of well-maintained all-weather gravel road from the tarred Letlhakane to Orapa road. Letlhakane village is the closest settlement and offers basic facilities. At the 2001 census Letlhakane had a population of 15,000 rising by 5.7% annually (Central Statistics Office, Gaborone), thus at present, probably has a population of 20,000 to 25,000. There are good telecommunications including cellular telephone networks in the area. Letlhakane is reached from the major cities of Gaborone and Francistown by good quality tarred roads. There is an 1800m airstrip at Karowe, however the closest airport with commercial flights is Francistown, some 200 km to the east and 2.5 hours away by road. There is also an airstrip within the nearby Debswana controlled Orapa Township.

5.3 Climate

The climate is hot and semi-arid, with an average annual rainfall of 462 mm at Francistown, which falls almost entirely in the summer months from October to April. Summer maximum temperatures are high, generally >30°C, whilst winter days are mild and the nights cold (often <10°C) with occasional ground frost. High diurnal ranges are experienced in all seasons. The climate does not impede mining operations, which can continue all year round. A summary of monthly average temperatures and rainfall are shown in Table 5.1.

Table 5.1 – Typical Climate and Rainfall

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave Temp Degrees C	24.6	24.0	23.0	20.7	17.1	14.2	14.1	16.8	21.1	24.6	24.9	24.5
Rainfall mm	80	72	46	25	2	1	0	0	5	23	46	63

5.4 Infrastructure and Local Resources

The area has a history of diamond mining dating back to 1971 when operations started at the nearby Orapa Mine, one of the largest diamond mines in the world. There is a reserve of qualified and experienced manpower in the immediate area. The major Ni-Cu mining operations at Tati Nickel, near Francistown, and at BCL, Selebi-Phikwe, have also added to the supply of labour with mining-related skills.

In terms of ML2008/6L, the Government supplies electrical power on commercial terms to the Karowe Mine through the Botswana Power Corporation's national grid.

Water for the existing diamond mines is derived from a strong aquifer at the contact of the Ntane Sandstone Formation and the overlying Karoo basalt. The Orapa, Letlhakane, and Damtshaa mines have a combined water demand of some 12M m^3/yr and this aquifer has successfully supplied the mines for over 40 years. The additional demand of approximately 2.6M m^3/yr from the Karowe Mine has been successfully met, and the aquifer remains robust.

Accommodation for personnel has been built by local companies and is leased by Boteti in Letlhakane.

6 History

The AK06 kimberlite was discovered by De Beers in 1969 during part of the same exploration programme that between 1967 and 1970 that discovered the Orapa kimberlite (named AK1) and the Letlhakane kimberlites (DK1 and DK2). This program also lead to a series of other kimberlite discoveries in the Orapa region.

Commercial production was achieved in July 2012 and has the mine has operated continuously since that date.

6.1 Early Work: De Beers Prospecting Botswana (Pty) Ltd and De Beers Botswana Mining Company (Pty) Ltd

De Beers Botswana Mining Company (Pty) Ltd. (the predecessor of the Debswana Diamond Mining Company (Pty) Ltd) held State Grant ("SG") 14/72 from 16 September 1972 until 15 December 1975. Under the grant, De Beers carried out evaluation and the delineation of kimberlites discovered previously. In addition they carried out reconnaissance and detailed soil sampling.

Little data from the initial discovery and evaluation of the AK6 kimberlite is available, but it is known that the discovery was made from the interpretation of an aeromagnetic survey. The kimberlite was delineated with 44 percussion boreholes, 20 of which were recorded as intersecting kimberlite and 24 as intersecting basalt. De Beers interpreted the AK6 kimberlite to have an area of 3.3 ha. A series of three 20 foot (~6.5 m) deep pits excavated in 1973 gave a grade of 0.07 ct/m^3 (approximately 3.5 ct/100t; this sampling was not NI 43-101 compliant).

One vertical cored borehole was drilled into the kimberlite to a depth of 61 metres with weathered primary kimberlite recorded from a depth of 8 metres (De Beers, 1976).

Reconstruction from the later exploration programmes suggests that two of the pits were sunk into basalt breccia, as were many of the percussion boreholes. There were two cored holes, as well as possibly two large diameter holes drilled with a jumper (cable tool) rig.

6.2 Debswana Diamond Company (Pty) Ltd. PL17/86

The current AK06 kimberlite and Karowe Mine lies within former PL 17/86 held by Debswana from 1 July 1986 until 24 January 1998. The kimberlite lies within the area dropped at the second relinquishment stage. The primary focus of the work programs on the license was to focus on the discovery of additional kimberlite intrusions however the AK06 was drilled for geological information and to test its diamond content (Debswana, 1999). No details of how it was drilled or sampled are provided, but it is stated as being 3.3 ha in area, comprising hard, dark green kimberlite breccia, and having a diamond grade of 0.42 ct/m^3 (approximately 15 ct/100t; not NI 43-101 compliant).

6.3 De Beers Prospecting Botswana (Pty) Ltd, PL1/97

PL1/97 was issued to De Beers Prospecting Botswana (Pty) Ltd. ("Debot") on 1 February 1997 and covered the AK06 kimberlite. However, the pipe was within the area dropped at first relinquishment in 2000, and no work was recorded on it.

6.4 De Beers Prospecting Botswana (Pty) Ltd, PL13/2000

In April 2000, Debot was granted PL13/2000 with an area of 9.95 km^2 over the AK06 kimberlite. Results from three small diameter percussion boreholes indicated the existence of the North and Central Lobes for the first time. The licence was renewed on 31 March 2003 with the area reduced to 4.90 km^2. In September 2003 De Beers carried out high resolution ground magnetic surveys over three kimberlites AK06, AK10 and BK11. The results of this work suggested that the AK6 kimberlite had a potential surface area of 9.5 ha, although much of this area was constituted of basalt breccia.

In December 2003, De Beers started a programme of five 12¼" boreholes intended to collect a 100 t bulk sample. The drilling was completed in February 2004, and the encouraging results only became available in October 2004, after the licence had been included in the Boteti Joint Venture.

6.5 The Boteti Joint Venture

On 17 April 2004, a joint venture heads of agreement was entered into between Kukama Mining and Exploration (Pty) Ltd and Debot for seven prospecting licences in the Orapa area totalling 1,344.27 km^2 and including 29 previously discovered kimberlites, which included kimberlite situated on PL13/2000. A twelve month work programme was carried out per the heads of agreement, which resulted in the signing of a formal joint venture agreement on 20 October 2004 and the incorporation of Boteti. Subsequently PL13/2000 was transferred to Boteti Exploration (PTY.) Ltd.

6.6 Boteti Exploration (Pty) Ltd and Boteti Mining (Pty) Ltd

The exploration work carried out by Debot on behalf of Boteti is described in Sections 9 to 12.

A Mining Licence application was submitted by the then Operator, Debot, on 28th September 2007. Previously, on 30th July 2007, Boteti had applied to the Government of Botswana under Section 25 of the Mines and Minerals Act for a Retention Licence over the AK6 kimberlite. On 9th September 2008, the Government informed Boteti that it would regard the period since the Retention Licence application as a negotiation period as allowed under Section 50 of the Act, and urged Boteti to apply for a Mining Licence. This was done, and ML2008/6L was issued effective from 28th October 2008.

On 24th May 2010, Boteti changed its name from “Boteti Exploration (Pty) Ltd” to Boteti Mining (Pty) Ltd.

6.6.1 Lucara Diamond Corporation

Lucara Diamond Corporation purchased a 70.268% interest in Boteti from Debot in November 2009 for US$49 million. Government approval which, under the Mines and Minerals Act Section 50 was a condition precedent for this transaction, was given on 18th December 2009. In April 2010, African Diamonds exercised its option to increase its interest by 10.268% at a cost of US$7.3 million. In addition, African Diamonds acquired Wati Ventures and its interest of 1.351% to bring their total shareholding in Boteti up to 40%.

In November 2010, Lucara and African Diamonds approved a plan for the construction of the Karowe Mine with full commissioning targeted for early 2012. On 20th December 2010, Lucara secured a 100% interest in the AK6 Project pursuant to an arrangement which combined the Company with African Diamonds Limited under a British court-approved scheme of arrangement.

On 25th July 2011, Lucara commenced trading its shares on the Botswana Stock Exchange, and on 29th August, Lucara commenced trading its shares on the TSX main exchange (after moving from the TSX Venture Exchange). On 25th November, Lucara commenced trading its shares on the NASDAQ OMX First North Exchange in Sweden.

In December 2011, the AK6 Project was renamed the Karowe Mine and construction of the mine was substantively completed by the end of March 2012 and the first production diamonds were recovered in April.

The commencement of full commercial production at the Karowe Mine was declared as of July 1, 2012 and by August 2012, the mine had ramped up to full production.

In November 2012, Lucara recovered a 9.46 carat ("ct") rare Type II blue diamond at Karowe Mine which it sold for US$4.5 million.

Since the onset of commercial production to the end of calendar year Q2 2017 the Karowe Mine/AK06 kimberlite has produced 2.0 million carats ("cts") from 12 million tonnes of processed kimberlite sold via tender a total of 1.87 million carats for a total of US$1.1 billion resulting in an achieved sold average price US$569/ct.

Table 6.1 - Production and Sales Results

Year	2012	2013	2014	2015	2016	Q1/Q2 2017
Kimberlite Mined (Tonnes)	1,600,971	3,944,343	3,327,754	2,358,657	2,722,375	563,397
Waste Mined (Tonnes)	4,074,196	5,493,445	10,270,720	11,407,010	11,058,041	5,579,373
Kimberlite Processed (Tonnes)	1,327,682	2,354,538	2,421,506	2,238,975	2,613,217	1,112,577
Carats recovered	294,167	440,751	430,292	365,690	353,974	122,865
Recovered Grade (cpht)	22	19	18	16	14	11
Carats Sold	152,724	438,717	412,136	377,136	358,806	126,879
Sales AP $/ct	$274	$415	$617	$612	$824	$852

In mid-November 2015 the Karowe mine recovered the World's second largest diamond gemstone, the 1109 carat Lesedi La Rona. The following day the Karowe mine recovered the 813 carat Constellation diamond. In addition to other diamonds of note, including the 342 ct "Queen of the Kalahari" the Karowe Mine is firmly established as one of the World's most significant producer of large and high-value diamonds, and has furthered Botswana's place at the forefront of global diamond mining.

Significant Stone recovery to end of June 2017

A total of 145 diamonds have sold for greater than US$1.0M a piece and over 73 gem quality diamonds greater than 100ct sold have been sold as individual stones. Since 2012 to end of Q2/17 the Karowe mine has recovered 7 diamonds >300 ct, 25 diamonds between 200 and 300 ct in weight and an additional 104 diamonds between 100 and 200 cts in weight.

7 Geological Setting and Mineralisation

A detailed account of the geological setting and geology of the Karowe Mine is given in the AK6 Kimberlite Technical Report dated March 25th 2010. A short summary is included here for reference.

7.1 Local and Regional Geology

The bedrock of the region is covered by a thin veneer of wind-blown Kalahari sand and exposure is very poor. Rocks close to surface are often extensively calcretised and silcretised due to prolonged exposure on a late Tertiary erosion surface (the African Surface) which approximates to the present day land surface.

The country rock at the Karowe Mine is sub-outcropping flood basalt of the Stormberg Lava Group which is underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup. The basalts, which are very extensive and underlie much of central Botswana, are Jurassic (180 Ma) and lie unconformably on the sedimentary succession, but are stratigraphically part of the Karoo Supergroup.

The regional stratigraphy is shown in Figure 7.1.

There are few outcrops in the Letlhakane area, as the bedrock is concealed by several metres of aeolian sand of the Kalahari Group, reflecting the area's position on the edge of the Tertiary Kalahari Basin. To the south and west of the Orapa Kimberlite Field, the bedrock may be overlain by up to 40 m of Kalahari Group sediments.

The Orapa Kimberlite Field lies on the northern edge of the Central Kalahari Karoo Basin along which the Karoo succession dips very gently to the SSW and off-laps against the Precambrian rocks which occur at shallow depth (although they are seldom actually exposed) within the Makgadikgadi Depression. The Karoo succession is condensed, with a total thickness of around 600 m, and is best preserved in WNW-ESE oriented grabens. The large AK1 kimberlite lies within such a graben (Coates et al., 1979).

The Orapa Kimberlite Field includes at least 83 kimberlite bodies, varying in size from insignificant dykes to the 110 ha AK1 kimberlite which is Debswana's Orapa Mine. All are of post-Karoo age. Of the 83 known kimberlite intrusions, five (AK1, BK9, DK1, DK2 and AK6 which is the Karowe Mine) have been, or are currently being mined, and a further four (BK1, BK11, BK12 and BK15) are recognized as potentially economic deposits.

Stratigraphic Unit			Lithologies
Supergroup	Group	Formation	
	Kalahari Group	Not differentiated in this area	Windblown sand, overlying duricrusts
~~unconformity~~~~~~~~~~~~~~~~			
			Kimberlite intrusions
~~unconformity~~~~~~~~~~~~~~~~			
Karoo Supergroup	Stormberg Lava Group (Drakensberg Group)		Very extensive flood basalts
~~unconformity~~~~~~~~~~~~~~~~			
Karoo Supergroup	Lebung Group	Ntane Sandstone Formation	Aeolian sandstone
		Mosolotsane Formation	Red mudstones (upper member), overlying red and green sandstones (lower member)
~~unconformity~~~~~~~~~~~~~~~~			
Karoo Supergroup	Ecca Group	Tlhabala Formation	Reddish grey non-carbonaceous siltstone, mudstone and shale. Weathers red, green or khaki
		Tlapana Formation	Black carbonaceous shale and coal
		Mea Arkose Formation	Coarse, white micaceous sandstone and dark shales
~~unconformity~~~~~~~~~~~~~~~~			
			Granite gneiss and amphibolite

Figure 7.1 – Stratigraphy

7.2 Property Geology

Drilling has shown the following country rock succession at the Karowe Mine property (Table 7.1). The volcanic and sedimentary units are almost flat lying.

Table 7.1 – Stratigraphic thickness recorded on the AK6 Project property

Depth from Surface	Stratigraphic Unit
Surface - ~8m	Kalahari Group
~8m – 135m	Karoo Basalt
135m – 255m	Lebung Group
255m – 360m	Tlhabala Formation
~360m - ~480m	Tlapana Formation
>480m	Granitic Basement

7.3 Kimberlite Geology

The geological summaries presented in this section, unless otherwise indicated, are extracted and summarized from internal De Beers documentation (Stiefenhofer, 2007; Hanekom et al., 2006; Tait and Maccelari, 2008). Mineral Services Canada has reviewed available relevant information, examined exposures of the kimberlite in the open pit and examined selected drill cores and is satisfied that the De Beers reports provide a reliable description of the geology of the AK6 body.

AK6 is a roughly north-south elongate kimberlite body with a near surface expression of ~3.3 ha and has a maximum area of approximately 7 ha at ~120 m below surface. The body comprises three geologically distinct, coalescing pipes that taper with depth into discrete roots. These "pipes" are referred to as the North Lobe, Centre Lobe, and South Lobe.

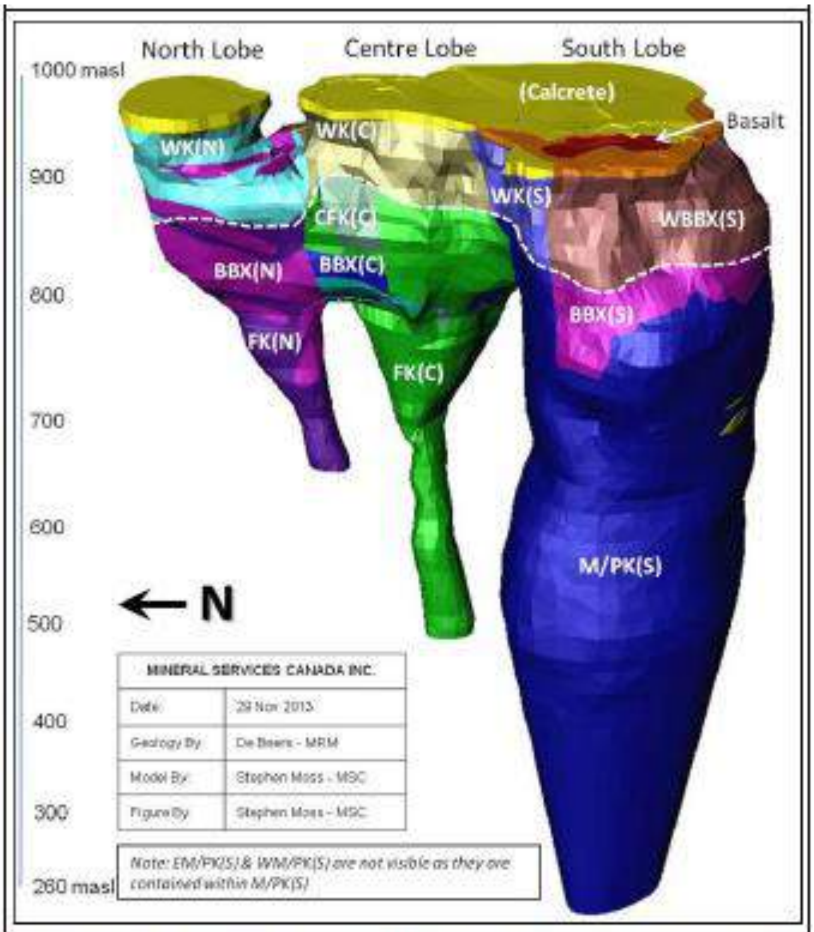
The AK6 kimberlite is an opaque-mineral-rich monticellite kimberlite, texturally classified primarily as fragmental volcaniclastic kimberlite with lesser macrocrystic hypabyssal facies kimberlite of the Group 1 variety (Field, 1989). The nature of the kimberlite differs between each lobe, with distinctions apparent in the textural characteristics, relative proportion of internal country-rock dilution, and degree or extent of weathering. The South Lobe is considered to be distinctly different from the North and Centre Lobes which are similar to each other in terms of their geological characteristics. The North and Centre Lobes exhibit significant textural complexity (reflected in apparent variations in degree of fragmentation and proportions of country-rock xenoliths) whereas the bulk of the South Lobe is more massive and internally homogeneous.

Kimberlite rock types identified in the AK6 kimberlite (encompassing true kimberlite as well as internal breccias dominated by basalt xenoliths) were grouped by De Beers into mappable units (Table 7.2) based on their geological characteristics and interpreted grade potential. This was based on extensive drill core logging supported by petrographic studies of representative samples, analysis and interpretation of groundmass spinel composition, and whole-rock geochemical analysis (Stiefenhofer and Hanekom, 2005; Hanekom et al., 2006; Tait and Maccelari, 2008). The main geological features of each unit are summarised below and their interpreted spatial distribution, as reflected in the geology model of AK6, is illustrated in Figure 7.2 and Figure 7.3. For the purposes of constructing the geological and grade model, each kimberlite unit occurring in each lobe was modelled as a separate geology solid, referred to in this report as a "domain" (Figure 7.2;Table 7.2).

Table 7.2 - Kimberlite units identified in AK6 kimberlite geological model

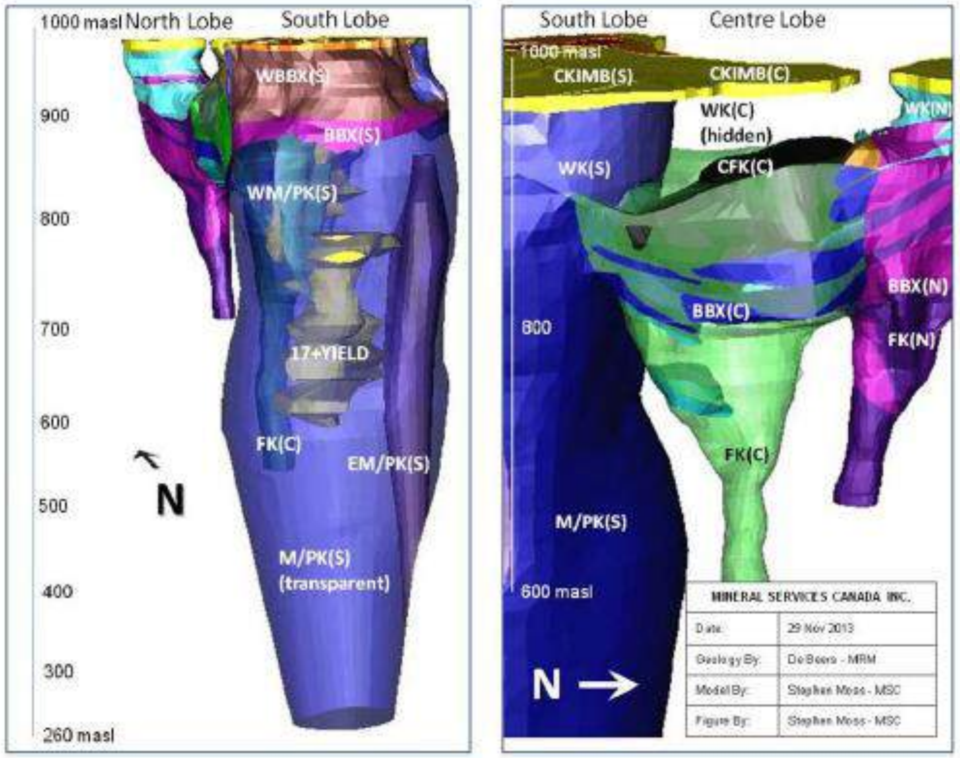
Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
North	CKIMB	CKIMB(N)	Calcretised kimberlite
North	FK(N)	FK(N)	Fragmental kimberlite
North	KBBX	KBBX(N)	Kimberlite and country rock breccia
North	WBBX	WBBX(N)	Weathered country rock breccia
North	WK(N)	WK(N)	Weathered kimberlite
Center	BBX	BBX(C)	Country rock breccia
Center	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
Center	CKIMB(C)	CKIMB(C)	Calcretised kimberlite
Center	FK(C)	FK(C)	Fragmental kimberlite
Center	KBBX	KBBX(C)	Kimberlite and country rock breccia
Center	WBBX	WBBX(C)	Weathered country rock breccia
Center	WK(C)	WK(C)	Weathered kimberlite
South	BBX(S)	BBX(S)	Country rock breccia
South	CBBX(S)	CBBX(S)	Calcretised country rock breccia
South	CKIMB(S)	CKIMB(S)	Calcretised kimberlite
South	EM/PK(S)	EM/PK(S)	Eastern magmatic/pyroclastic kimberlite
South	INTBS	INTBS(S)	Large internal block of basalt
South	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite
South	M/PK(S)	17+YIELD	High-yield magmatic/pyroclastic kimberlite
South	WBBX(S)	WBBX(S)	Weathered country rock breccia
South	WK(S)	WK(S)	Weathered kimberlite
South	WM/PK(S)	WM/PK(S)	Western magmatic/pyroclastic kimberlite

Units occurring in more than one lobe (e.g. BBX, CKIMB) were modelled as separate domains for each lobe. The 17+YIELD material identified in the South Lobe is a variety of M/PK(S) kimberlite defined as a separate domain on account of its very high concentrate yield.



Inclined profile view of the geology model of the AK6 kimberlite. Geology domain names are indicated in white lettering. Dashed line indicates the maximum depth of the weathering horizon. Geology model modified after original De Beers model (Opperman and van der Schyff, 2007; see Section 14.1).

Figure 7.2 - Geological Model of AK6 Kimberlite



Inclined profile views of the geology model of the AK6 kimberlite highlighting the internal domains within the South Lobe (left panel) and Centre Lobe (right panel). Geology model modified after original De Beers model (Opperman and van der Schyff, 2007; see Section 14.1).

Figure 7.3 - Inclined profile views of the AK6 geological model highlighting internal domains

7.3.1 Calcretised kimberlite (CKIMB)

The upper parts of all three lobes contain severely calcretised and silcretised rock. This zone is typically approximately 10 m in thickness, extending up to 20 m in places. Due to the destruction of textures and resultant difficulty in recognizing specific lithologies within this zone, it has been modelled as a separate single unit extending across all three lobes (Opperman and van der Schyff, 2007). For Mineral Resource modelling purposes, the CKIMB unit was divided by lobe into three separate domains.

7.3.2 Weathered kimberlite (WK)

The upper 30 to 50 m of the kimberlite is highly weathered. The intensity of weathering decreases with depth with fresh kimberlite generally intersected at about 70 to 90 m below present day surface. Although the primary mineralogical and textural features of the kimberlite are obscured in the upper portions of the weathered zone, this material is seen to transition into the underlying fresh kimberlite units in each lobe. Thus separate weathered units have been defined in each lobe for each of the geology domains where weathered equivalents of these domains are present at surface (Table 7.2; Figure 7.2 and Figure 7.3). Separate models of these units are required as weathering has a significant impact on the percentage DMS yield as well as on the density of the kimberlite.

7.3.3 Basalt breccia (BBX/KBBX)

Each of the lobes is characterised by discontinuous zones of brecciated basalt (BBX), mixed with variable, but generally small amounts of kimberlite, typically less than 10%. The basalt breccias consist of large (meter-sized) to smaller basalt clasts set in a matrix of kimberlite. Basalt clasts are variably fractured and carbonate-veined and consist of vesicular and non-vesicular varieties.

Opperman and van der Schyff (2007) only modelled the largest of these breccia zones and indicated that small zones of basalt breccia may be encountered in each of the pipes during mining or further evaluation. Given the density of coverage of available delineation drilling, however, large areas of high dilution that would significantly impact overall diamond grade are unlikely to be present. The bulk of the breccias occur close to the wall-rock contacts in each lobe. Basalt breccia represents a single geological unit, but for the Mineral Resource update documented in this report it has been separated into different domains by lobe (Table 7.2). An additional geology unit (KBBX) was defined to encompass kimberlite breccias that are broadly similar to the BBX described above, but display lower levels of country-rock dilution (50% to 90%). KBBX zones appear to be interbedded and/or spatially associated with BBX domains. Tait and Maccelari (2008) interpreted KBBX deposits as either talus-type slump deposits or as deposits of possible pyroclastic origin (given their higher kimberlite content relative to BBX).

7.3.4 North Lobe Kimberlite Units

FK(N) – Fragmental kimberlite (North Lobe)

The North Lobe is predominantly infilled by a light greenish-grey, medium-grained (4 to 32 mm), matrix-supported, poorly sorted, massive fragmental volcaniclastic to superficially magmatic kimberlite (Hanekom et al., 2006). Basalt is the dominant country-rock xenolith type with lesser basement and Karoo sedimentary rock fragments. Two broad textural groups in the kimberlite of the North Lobe were identified: rocks with a matrix consisting of both serpentine and calcite, and samples with a matrix consisting predominantly of serpentine with minor calcite. No clear spatial distinction between the two groups could be resolved and the fragmental kimberlite was modelled as a single unit and domain.

7.3.5 Centre Lobe kimberlite units

The Centre Lobe is infilled by kimberlite that bears a superficial resemblance to the kimberlite from the North Lobe in that both lobes include non-fragmental, apparent magmatic material as well as fragmental volcaniclastic kimberlite (Hanekom et al., 2006). Macroscopically, colour and texture variations are common within Centre Lobe, but contacts between texturally distinct zones are generally gradational. Kimberlite textures locally alternate between superficially nonfragmental and more fragmental (volcaniclastic), similar to that of the North Lobe. The most consistent recognisable difference between the Centre Lobe and North Lobe kimberlite infill is a higher carbonate content in some samples from the Centre Lobe relative to North Lobe. Two main units of fresh kimberlite are recognised in the Centre Lobe.

CFK(C) – Carbonate-rich fragmental kimberlite (Centre Lobe)

The fresh infill in the upper part of Centre Lobe comprises a medium-grained (4 to 32 mm),matrix-supported, poorly-sorted and massive, carbonate-rich fragmental kimberlite. Basalt represents the dominant country-rock xenolith type with lesser basement and Karoo sedimentary rock fragments present. Microscopically, the majority of samples show carbonate infilling of voidspace, highlighting the potential fragmental texture of the kimberlite. Point counting data reported by Hanekom et al. (2006) on a very limited sample suite suggest that the carbonate-rich fragmental kimberlite generally contains higher concentrations of olivine macrocrysts and lower country-rock xenolith concentrations than those of the fragmental kimberlite unit (see FK(C) – Fragmental kimberlite (Centre Lobe) below). The groundmass opaque-mineral content is also slightly higher, although overlap occurs.

FK(C) – Fragmental kimberlite (Centre Lobe)

The remaining fresh kimberlite within the Centre Lobe comprises matrix-supported, poorly sorted and massive fragmental kimberlite which is distinct from CFK(C) due to an apparent relative decrease in carbonate content. Hanekom et al., (2006) noted that samples showing clay alteration and thin magmatic selvages around olivine grains and country-rock xenoliths, i.e. a more volcaniclastic appearance, are generally but not exclusively associated with areas of increased country-rock xenolith content. This material is often greenish in colour and characterised by the presence of large blocks of basalt. Basalt breccia units in the Centre Lobe also occur within the fragmental kimberlite unit rather than in the carbonate-rich fragmental kimberlite unit. Basalt represents the dominant country-rock xenolith type with lesser basement and Karoo sedimentary rock fragments.

7.3.6 South Lobe kimberlite units

Previous reports summarizing detailed drill core logging, petrographic analyses, and geochemical analyses demonstrate the distinct character of the South Lobe kimberlite in comparison to that of the North and Centre Lobes (Hanekom et al., 2006; Stiefenhofer, 2007; Stiefenhofer and Hanekom, 2005). The upper, western part of the South Lobe is dominated by a weathered basalt breccia (WBBX(S)) with an underlying unaltered basalt breccia unit (BBX(S)), and also includes a large block (“floating reef”) of solid basalt recently recognised and mapped during mining activities.

M/PK(S) – Magmatic/pyroclastic kimberlite – South Lobe

The South Lobe is dominantly infilled by medium–grained to coarse (4 to >32 mm), matrix supported, poorly-sorted and massive, macrocrystic magmatic/pyroclastic kimberlite. The name of this unit reflects the initial uncertainty with respect to the textural classification of the kimberlite. The kimberlite exhibits textures consistent with a magmatic or hypabyssal kimberlite (HK), but also exhibits subtle textures suggesting a possible pyroclastic origin (PK). Macroscopically the kimberlite is grey in colour and contains approximately 5% to 10% thermally metasomatised/altered country-rock xenoliths. Olivine grains are relatively fresh and abundant opaque minerals are present. Fresh monticellite is present and increases in abundance with depth.

Country-rock xenoliths are predominantly basalt with lesser basement and Karoo sedimentary rocks, but the overall proportion of crustal dilution is very low (typically <10%), rarely ranging up to a maximum of 25%. Minor zones of crude layering are locally apparent, defined by accumulations of olivine macrocrysts and sub-horizontal preferentially oriented crustal xenoliths. These zones range from 0.16 m to 1.5 m in thickness.

EM/PK(S) – Eastern diluted magmatic/pyroclastic kimberlite – South Lobe

A pipe-shaped internal kimberlite unit, defined by De Beers along the eastern part of the South Lobe, comprises coarse to medium-grained (4 to >32 mm), matrix-supported, poorly sorted and largely massive magmatic kimberlite which is distinct from M/PK(S) primarily due to an apparent increase in small (typically <1 cm) country-rock fragments, readily visible in drill core. Hanekom et al. (2006) reported that this unit contains fewer olivine macrocrysts in comparison with the remainder of the South Lobe and abundant coarse microlitic diopside was observed in thin section. Perovskite appears to be slightly more abundant in the diluted zones and the groundmass shows a greenish colour, possibly due to serpentinisation. Country-rock clasts primarily comprise basalt, with less common xenoliths of basement, and Karoo sedimentary rock. Basement fragments may locally be more abundant than in the M/PK(S) unit. Proportions of crustal dilution range from 3% to 10%. Greenish serpentinised zones are common. In addition to the visual differences, EM/PK(S) exhibits differences in whole rock geochemistry, percentage DMS yield and bulk density relative to M/PK(S).

WM/PK(S) – Western diluted magmatic/pyroclastic kimberlite – South Lobe

A pipe-shaped internal kimberlite unit has also been defined in the western portion of the South Lobe and displays geological characteristics that appear different to those of the M/PK(S) and EM/PK(S) units. WM/PK(S) comprises greenish-grey, medium-grained (4 to >32 mm), matrix supported, poorly sorted, massive magmatic kimberlite, and is macroscopically distinct in colour due to its apparent altered character. This material shows additional differences in whole rock geochemistry, percentage DMS yield and rock density relative to EM/PK(S) and M/PK(S). Olivine is serpentinised and locally completely weathered out from drill core. Basalt represents the dominant country-rock lithology. Less common basement and rare black shale xenoliths are also present in places. Crustal dilution ranges from 7% to 36%. The geometry of this unit is somewhat speculative due to sparse drill coverage.

17+YIELD

Metallurgical testing and processing of large diameter drill samples indicated a substantial variation in the ratio of concentrate yield to wet head feed mass (Stiefenhofer, 2007). A cut-off value of 17% DMS yield was applied to large diameter drill samples, and the resulting envelope around these high yield samples forms a sub-zone within the interior core of the M/PK(S) unit that was modelled as a separate domain due to its metallurgical significance for processing. This solid is thus not a traditional geological boundary but rather an envelope defining a percentage yield value. The substantial variation in yield throughout the South Lobe is attributed to a combination of primary and secondary processes, including olivine concentration, alteration and country-rock dilution.

8 Deposit Types

There are two main varieties of diamonds deposits, those that are primary deposits and secondary deposits. The primary deposits are those where diamonds remain within the host rock (kimberlite, lamproite are main host rocks). Secondary diamond deposits (alluvial) result from the erosion of diamond from the primary host and concentration via wind or water into an alluvial deposit.

The AK06 kimberlite at the Karowe mine is a primary diamond deposit. The intrusion is a kimberlite diatreme, constituted of three distinct lobes.

Kimberlites are hybrid mantle-derived ultramafic magmas (>150 km source depth) that commonly entrain and transport rocks and minerals, through which the kimberlite magma passes, during ascent. This entrained material may consist of mantle, crustal or earlier kimberlite magmatic events exploiting the same intrusive network. Kimberlite may also entrain and transport diamonds together with the rocks from which the diamonds are directly derived (of peridotite and eclogite origin) to the earth's surface (e.g. Mitchell, 1986; Mitchell1995). Diamonds are not a kimberlite rock forming mineral but rather a xenocryst or foreign object within the kimberlite.

There are two varieties of kimberlite: Group 1 and Group 2. The Group 1 kimberlite magmas are typically dominated by olivine set in a fine-grained matrix commonly rich in serpentine and/or carbonate as well as varying amounts of phlogopite, monticellite, melilite, perovskite and spinel (chromite to titanomagnetite) and a range of accessory minerals (Mitchell, 1995). AK06 is a Group 1 kimberlite intrusion.

While some olivine crystallises directly from the kimberlite magma on emplacement (to form phenocrysts), kimberlites generally include a significant mantle-derived (xenocrystic) olivine component that typically manifest as large (>1 mm) rounded crystals. In addition to olivine, kimberlites also commonly contain significant quantities of other mantle derived minerals, the most common and important being garnet, Cr-diopside, chromite and ilmenite. These minerals, commonly referred to as indicator minerals, are important for kimberlite exploration and evaluation as they can be used both to find kimberlites (by tracing indicator minerals in surface samples) and to provide early indications of their potential to contain diamonds (Nowicki et al., 2007; Cookenboo and Grütter, 2010).

Diamonds are a high pressure (~50 Kbar) and temperature (~1,200°C) variety of carbon, which form at depths of at least 150 km below the earth's surface.

Only a small minority of kimberlite bodies contain diamonds in sufficient concentrations to be considered as diamond ore or economical to mine. Those which do have elevated diamond contents usually occur in areas underlain by old and stable crust, which are typically found in the cratonic cores of continental blocks. Cratonic areas are characterised by thick crust and low geothermal gradients. Indicator mineral chemistry is used as a guide to determine depth of source and potential for diamond entrainment. The transportation of entrained diamonds to the surface must be rapid in order to prevent their resorption or retrogression to graphite as pressure is released.

In economically viable deposits, diamonds are usually present in extremely small quantities, and their distribution within the host tends to be erratic (e.g. a grade of 20 carats per hundred tonnes (cpht) is equivalent to 0.04 parts per million)

The size and value of stones is erratic and it is possible that the bulk of the value of a parcel of diamonds is attributable to a small number of individual stones or even a single stone. Diamond liberation and recovery during sampling work must be monitored and may be impacted by the selection of drilling method, kimberlite hardness, and the selected recovery methodologies. The internal geology of kimberlites may be variable and as such the diamond content and size frequency distribution may varying within the intrusion by rock type. A well constrained geological model is required as well as a spatially representative sampling campaign designed to accommodate the distribution of different rock types.

In order to determine the typical revenues to be expected for a diamond deposit, for each of the identified and mappable units within the kimberlite the grade (carats per m^3, or carats/tonne), dilution, diamond size frequency distribution, and a diamond price model are all required. The diamond price model is a function of the size distribution by diamond size class and diamond value. Diamond value is determined by the valuation of rough parcel diamond parcels recovered from sampling of the various units within the kimberlite being evaluated. Rough diamond price is dependent on the physical characteristics of the individual diamonds within the parcel (weight, colour, clarity, shape, fluorescence). Large parcels of diamonds are better suited to provide an adequate view on diamond price. Most evaluation parcels are not of sufficient weight to fully represent the full diamond population from an SFD or value perspective, therefore SFD and value models are generated. Models will mitigate the impact of outlier diamonds, whether they be outliers due to weight (size) or value.

Diamonds are classified into two general groups based on the presence or absence of nitrogen impurities (Breeding and Shigley, 2009). Type I diamonds are nitrogen bearing (measureable by IR spectroscopy) and Type II diamonds which don not contain easily detectable nitrogen (Breeding and Shigley, 2009). Both Type I and Type II diamonds are further broken into additional subgroups based on how the nitrogen or other impurities (i.e. boron) are distributed within the crystal lattice of the diamond. Type IIa diamonds, due to their lack of impurities may be colourless and often command higher prices for the rough and the polished goods. Type IIa diamonds may be highly resorbed without primary crystal faces, elongated to flat and may be white (colourless) or brownish to grey. Blue Type II diamonds are related to boron impurities within the crystal structure. Type IIa diamonds only rarely display faint fluorescence in ultra-violet light. Due to the low nitrogen contents Type IIa diamonds may be of top colour (D, E, F).

Type IIa diamonds are rare in nature with few kimberlites producing large Type IIa. The Premier kimberlite in RSA has produced the world's largest Type IIa, the Cullinan, at 3,106 carats, the second largest Type IIa diamond recovered, the Lesedi La Rona, is from the Karowe Mine (AK06 kimberlite) and weighs 1109 carats. The Karowe mine has also produced the Constellation (813 carats), the world 6 largest gem quality diamond. Other regions known for producing Type IIa diamonds of significance are Lesotho (Letseng and Mothae kimberlites) and deposits in western Africa (Angola, Sierra Leone). The AK06 kimberlite produces a relatively high proportion of Type IIa diamonds throughout the size classes, especially within the +10.8ct sizes, and as a result Karowe diamonds are of a high average value.

9 Exploration

This section summarizes advanced exploration work on the AK06 kimberlite done by Boteti Exploration (Pty) Ltd from December 2003 until the completion of the final geological report in May 2007. All work was carried out by De Beers Prospecting Botswana (Pty) Ltd, the operator of the Boteti joint venture, under PL13/2000. Details on previous work programs are summarised here and detailed in Lynn et al., 2014, McGeorge et al, 2010 and the various references within. The current resource model is based mainly upon data collected during these programs with modifications resulting from mining operations and diamond sales since 2012 (Lynn et al., 2014; this report).

The AK06 kimberlite was continuously held by De Beers under a succession of prospecting licences from the time of its discovery in 1969, until the project was acquired by Lucara in 2008. The historic, limited and shallow, sampling had shown that it was diamondiferous, but it was initially thought to be very low grade and relatively small (3.3 ha) and as a result further exploration was not a priority. Subsequent work documented a basalt breccia around and over parts of the kimberlite, which was not fully appreciated early in the exploration history of the resource, and that the resource was previously under-sampled.

9.1 Exploration Approach and Methodology

The exploration of AK6 kimberlite followed a staged approach, which can be summarized as follows:

Initial exploration work – prior to the Boteti Joint Venture, in late 2003, De Beers carried out geophysical surveys and drilled 5 x 12¼" holes, which gave a 97 t (in-situ) bulk sample. This resulted in a sampling grade of ~23 ct/100 t and good quality diamonds. Due to a 10-month lapse between the completion of drilling and the release of the sampling results, De Beers committed PL13/2000 to the Boteti Joint Venture prior to these encouraging results being known.

Advanced Exploration Phase 1 – Based on the initial work, the AK6 kimberlite was declared an "advanced exploration project". The next step was to define a high confidence Inferred Mineral Resource and recover 500 ct from 13 large diameter drill holes at 70 m spacing. The external contacts and internal geology of the kimberlite were explored through an extensive programme of delineation drilling and high resolution geophysics.

Advanced Exploration Phase 2 – the results of phase 1 merited phase 2, the objective of which was to define an Indicated Mineral Resource and recover a large diamond parcel, ideally 3,000 ct, to reduce revenue uncertainty. Large diameter drill holes were placed at 50 m centres and trenches prepared for recovery of the required parcel of diamonds. Further delineation drilling was also done.

Advanced Phases 1 and 2 overlapped in time, due to a decision to fast track the project.

Initial conceptual mining studies showed that exploration should extend to 400 m below surface in the South Lobe, and 250 m below surface in the North and Central Lobes. These were the limits of possible open pit mining based on an initial economic assessment.

In 2016/17 two core drilling programs were conducted on the AK06 kimberlite. The combined drilled metres of 12624 provided additional pierce points and geological information for the south lobe between 400 and 600 metres below surface. Production and diamond sales results for South Lobe kimberlite exceeded that of the original 2014 resource models, in particular the coarse SFD of the south Lobe and the high achieved dollar per carat. The results of the drill programme, and the implications to the overall resource model are still pending. Drill data for the resource expansion program are outlined in Table 9.1 below. The drilling programmes are discussed in detail in Section 11 below.

Table 9.1 – Summary of Exploration Programme

Stage	Work done	Duration
100 tonne bulk sample	5 x 12¼" large diameter drill holes totaling 679 m.	2003 - 2005
	DMS and diamond recovery	
	geophysical surveys	
Advanced Exploration Program Phase 1	44 x 6½" percussion holes for delineation totaling 4,575 m	2005 - 2006
	12 x cored boreholes (NQ) as LDD pilots, totaling 2,980 m	
	17 x inclined boreholes (NQ) for delineation totaling 6,904 m	
	13 x 23" LDD totaling 3,699 m	
	DMS processing of 1,775 tonnes	
	diamond recovery from 112 tonnes of concentrate	
Advanced Exploration Program Phase 2	11 x cored boreholes (NQ) as LDD pilots totaling 4,181 m	2006 - 2008
	29 x inclined boreholes (NQ) for delineation totaling 8,679 m	
	12 x 23" LDD totaling 4,265 m	
	DMS processing of 2,235 tonnes	
	diamond recovery from 194 tonnes of concentrate	
	trenching	
	DMS processing	
	diamond recovery	

9.2 Geophysical Surveys

The AK6 kimberlite was first detected from an aeromagnetic survey in 1969. During 2005, the kimberlite was surveyed by DeBeers in great detail by four ground geophysical methods as outlined in Table 9.2 below. The geophysical data was used in the preparation of the first geological model and in volume calculations.

The geophysical surveys were highly effective in delineating the kimberlite. However drilling results show that the geophysical interpretations lead to an overestimation of the surface area of the kimberlite, since the surveys interpreted associated basalt breccias as "kimberlite". This over-estimation has been subsequently resolved by detailed drilling.

Table 9.2 - Summary of high resolution geophysical surveys conducted over the AK6 Kimberlite

Method	Line km	Comments
Magnetics	262.4	Very strong positive magnetic response, possibly influenced by basalt content
Gravity	62.6	Complex anomaly but overall a subtle Bouguer gravity negative due to the weathering of the pipe
Electro-magnetics (Geonics EM34 frequency domain)	57.6	Approximately defines kimberlite contacts
Controlled Source Audio-frequency Magneto-Tellurics (CSAMT)		Detected the three lobes at depth

10 Drilling

Drilling of the AK6 kimberlite is described in detail in a previous technical report dated 25th March 2010 (McGeorge et al., 2010) and references therein. A brief summary is provided here.

Beginning in late 2003, extensive drilling works were undertaken on the AK6 kimberlite. The drilling can be divided into that done to delineate the extent of the kimberlite and to map its internal geology, and density, and that done to obtain large kimberlite samples for diamond grade and revenue estimation. The drilling is summarized in Table 10.1, grouped into the exploration phases described in Section 9 above. Borehole locations are illustrated in Figure 10.1.

In 2016/17 two drill programs were conducted to test the deeper portions of the south lobe below 600 mamsl and provide geotechnical information on host rock stratigraphy. The combined metres drilled for the GT (3 holes) and KREP (12 completed, 1 abandoned holes) was 12624 metres.

Table 10.1 - Summary of exploration drilling programmes on the AK6 Kimberlite

Phase of programme	Purpose of drilling	Drill Type	Hole size	No. holes	Total metres	Duration
100 tonne bulk sample	Initial sampling	percussion (reverse circulation)	12¼"	5	679	late 2003-2/2004
Advanced Exploration Program Phase 1	delineation	percussion	6½"	44	4,575	2004-2005
	delineation	core	NQ	17	6,904	2/2005-10/2005
	piloting	core	NQ	12	2,979	
	bulk sampling	LDD	23"	13	3,699	7/2005-2/2006
Advanced Exploration Program Phase 2	piloting	core	NQ	11	4,181	11/2005-08/2006
	delineation	core	NQ	29	8,679	04/2006-02/2007
	bulk sampling	LDD	23"	12	4,265	04/2006-08/2006

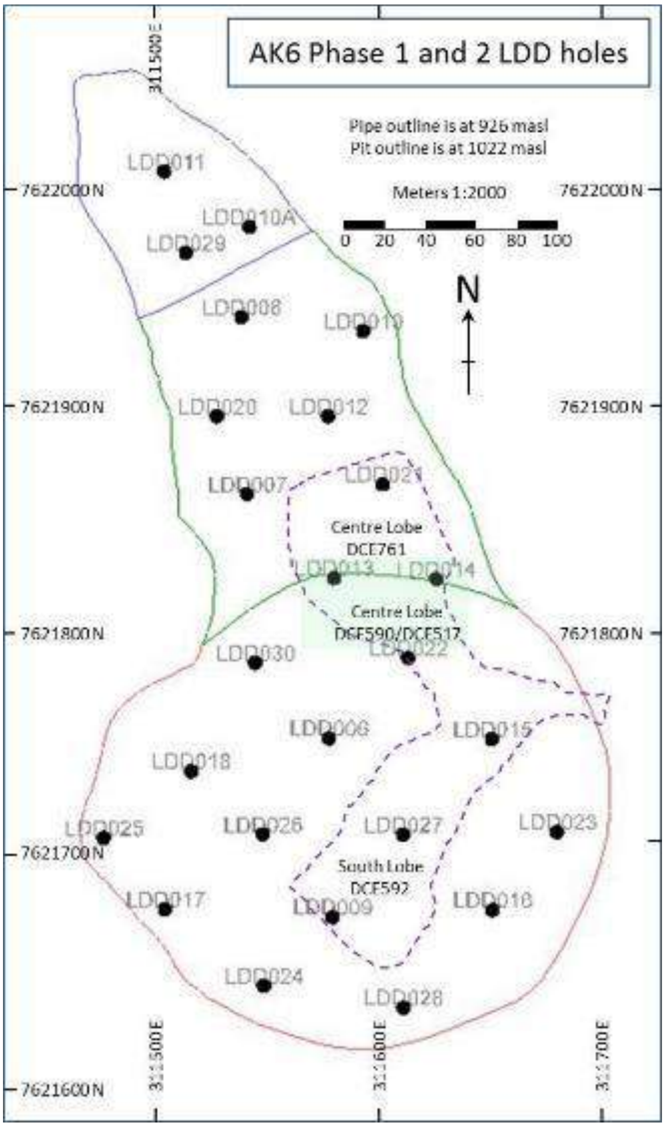


Figure 10.1 – Map of LDD Holes and Bulk Sample Trench (May 2006)

11 Sample Preparation, Analyses and Security

The sample preparation, analyses and security measures applied during the original evaluation programme is described in the previous technical reports and is summarised here for reference.

Samples for macro diamonds (+1.0 mm diamonds) were taken by means of:

- Five 12¼" reverse circulation boreholes.
- Twenty five 23" reverse flood airlift assist drilling.
- Trenching, where sample was loosened by blasting and/or earthmoving equipment.

All sample preparation and analysis was done by De Beers between 2003 and 2007.

11.1 Reverse Circulation, 12¼" Drilling

The sample returned by the drill was de-slimed on site, using a 1.47 mm screen. No details of the quality control applied to this sample collection are available. For example, it is not known whether checks were made for +1.47 mm material passing into the undersize.

All the material from each borehole was combined and treated as a single sample. The samples were sent to the De Beers Evaluation Services Department ("ESD") laboratory in Kimberley, South Africa, and a concentrate prepared using a DMS plant. No details of this work, or of quality control measures, are provided. The concentrate was shipped to the De Beers Group Exploration Macro-Diamond Laboratory ("GEMDL") in Johannesburg for diamond recovery. Due to inadequate information, it is not possible to comment on the sample preparation, security and analytical procedures.

11.2 Reverse Flood, 12¼" Drilling

There were two phases of Large Diameter Drilling ("LDD") work, the first from July 2005 – February 2006 (13 holes, 3,699 m) and the second from April 2006 – August 2006 (12 holes, 4,265 m). Sample preparation and analysis procedures were the same for both phases. Sample returned from the drill was de-slimed to +1.0 mm at the drill using a vibrating screen. The undersize screen was monitored for loss of +1.0 mm material, and if observed, the drill was stopped until the problem was addressed.

The sample was collected from the screen in cubic meter sample bags, under supervision of a geologist. It was then transported to the DMS plant at the De Beers Letlhakane camp on a flatbed lorry, also under the charge of the geologist. At the camp, the responsibility for the sample passed to the plant foreman.

The plant was a 10 t/hr dense media separation mobile unit. During phase 1, the plant received 1,775 t of headfeed, which produced 112 t of concentrate, giving an average DMS yield of 6%. Samples from the South Lobe had a significantly higher yield (7.8%) than those from the Central and North Lobes (mean 1.1%). This can be related to the higher density of the South Lobe (average 2.78 t/m^3) against the North and Central Lobes (average 2.43 t/m^3). During phase 2, the total headfeed was 2,235 t which produced 194 t of concentrate, giving a yield of 8%.

Following processing, the concentrates were collected in plastic drums which were sealed with security tags, and stored within a secure cage. The drums were then placed in sea containers with infra-red motion detector surveillance. Concentrates were shipped to GEMDL in Johannesburg inside sealed shipping containers that were carried on flatbed trucks. The loading of the trucks was supervised by Debswana security and the Letlhakane police. Both Debswana security and the Letlhakane police escorted the trucks to the Botswana / South Africa border. Once cleared through customs, the trucks were escorted within South Africa by De Beers security officials. The documentation accompanying the concentrates was in accordance with the Kimberley Process.

11.3 Trench Samples

The trench samples were concentrated at the Letlhakane camp in a similar manner to the LDD samples, except that in order to reduce the volume of sample to be processed through the plant and GEMDL, part of the sample was treated with a +2.00 mm bottom screen cut-off.

Coarse +6.0 mm tailings from the DMS plant were re-crushed to -6.0 mm and resubmitted to capture diamonds locked in the larger size fraction. Undersize tailings (+1-6 mm) were discarded. The trench material required some modifications to the plant, which also allowed the average feed rate to be increased to 12 t/hr. The modifications were:

- Mobile jaw crusher pre-crushed the trench samples to -100 mm.
- A tertiary scrubber was added to reduce fines.
- The secondary jaw crusher was replaced by a cone crusher in an attempt to reduce Diamond breakage.
- Installation of a tailings screen and conveyors.
- Installation of a flocculant addition system at the de-sliming cyclone.
- Replacement of the 200 mm degritting cyclone with a 350 mm degritting cyclone.

Sample was taken to the Letlhakane camp in haul trucks owned by the mining contractor Strata Mining (Pty) Ltd. The samples were stockpiled within the camp security area and each pile marked with a metal tag bearing the sample number. The concentrates were collected and shipped in the manner described above for the LDD samples. Their treatment at GEMDL was as for the LDD samples.

11.4 Diamond Recovery

Diamond recovery for the purposes of grade estimation, was done at GEMDL in Johannesburg. The diamond recovery parameters at GEMDL were the same for all phases. The GEMDL facility is fully ISO17025 certified. The recovery area of the GEMDL is a security “red area” and is subject to access control, three tier surveillance and hands off processing.

The concentrates arrived at GEMDL in the same sealed 50 litre drums they had left the sample plant in. Samples weighing 10 kg or more (wet) were treated through the main processing section. Drums within one specific sample were combined to expedite treatment and ease of handling. Material of -4 mm was passed through a dry X-ray sorting process with subsequent magnetic scalping of the X-ray tails to recover non-luminescent diamonds. Material +4 mm was passed through a wet X-ray process with the X-ray tailings dispatched as process tailings.

Diamond sorters removed diamonds from the prepared sample fractions. This was done inside secure glove boxes and recovered diamonds were placed into magnetically sealed diamond canisters.

All of the X-ray concentrates were sorted three times, and non-magnetic fractions were sorted once or twice. The sorting efficiency was set at 98% diamond recovery (per carat weight). Recovered diamonds were sent to the final sorting section and stripped concentrate tailings to the hand sort tailings packaging section. Final sorting consisted of a number of processes aimed at arriving at a DTC sieve class for each sample. There was also a de-falsification process which involves the removal of miss-identified material which is not diamond. If necessary an infra-red spectrometer was used to confirm diamond.

All equipment and floors were purged between consignments. For quality assurance, monitor diamonds were added to the sample by an external monitoring team. After defalsification, the monitor diamonds were removed.

The diamonds were then sieved using DTC standard diamond sieves. Larger diamonds (+ 3 sieve) were photographed. Diamond breakage studies were done on a selection of diamonds to estimate the amount of diamond damage on individual stones from the drilling and sample treatment processes. The impact of diamond damage on the size frequency distribution can be offset in the resource modelling process, by reference to microdiamond data. However, the impact on revenue estimation is more difficult to address.

The diamonds were then sent to Harry Oppenheimer House in Kimberley, for acid cleaning, re-sieving and final re-weighing. The X-ray tailings were reconstituted and put into 50 litres blue plastic drums, packed into 6 m shipping containers, and returned to site.

12 Data Verification

The geological, bulk density and grade data that form the basis for the Mineral Resource Estimate documented in this report are unchanged from those reported in Lynn et al. (2014). Hence Sections 12.1 to 12.6 are restated directly from Lynne et al. (2014). Section 12.7, relating to the production data on which diamond value estimates have been based has been updated for the purposes of this report.

A mineral resource estimate for the AK6 kimberlite pipe extending to a depth of 750 mbs (255 mamsl) was initially undertaken by De Beers in 2006 (Phase 1; Bush, 2006), and updated in 2007 to reflect changes to the geology model (Phase 2; Bush, 2007). A number of internal and external audits and assurance reviews were undertaken by various parties to determine the quality of data used to develop the AK6 mineral resource model. These are summarized in McGeorge et al. (2010). The mineral resource estimate was reviewed internally by the Mineral Resource Management (MRM) group at De Beers (De Beers MRM, 2006; De Beers MRM, 2007; Bosma, 2008) and the 2007 estimate was further reviewed externally by Z-Star (Bush, 2008a; Bush, 2008b) and Anglo American (Rice et al., 2008).

The review of the mineral resource estimate by Z-Star resulted in a small net decrease in the resource volume (1.3%) and total carats (0.35%). The independent review by Anglo American identified and corrected errors in the geological data, and resulted in a description of the geology model as coherent and robust, and the mineral resource estimate as comprehensive. The biggest risk identified was the revenue model, due to the relatively small parcel of diamonds valued at the time (McGeorge et al., 2010).

The De Beers mineral resource estimate, incorporating the results of internal and external reviews, was subsequently incorporated into a Feasibility Study reported by MSA in 2010 (McGeorge et al., 2010). Minimal further verification of the technical work was possible for the 2010 report due to the long interval of time between completion of the work and when the report was commissioned (McGeorge et al., 2010).

The focus of the sections below is on verifying critical aspects of the previously reported mineral resource estimate (McGeorge et al., 2010), as well as of production and geological data obtained subsequent to 2010. Updates or modifications of the mineral resource estimate for AK6 required as a result of new information obtained subsequent to 2010 are discussed further in Section 14.

12.1 Basis for Geological Model

The different kimberlite units identified within each lobe of AK6 (see Section 7.3) were initially determined by De Beers on the basis of visual criteria, and subsequently supported using petrography and geochemistry data. The units identified to date represent either distinct kimberlite types with the potential for different diamond populations and/or diamond grade (e.g. M/PK(S), FK(N), FK(C)), or metallurgically-distinct zones such as “weathered” kimberlite (e.g. WK(S); WK(C); WK(N)) which have potential implications for mining and processing.

Petrography reports (Stiefenhofer, 2007; Field, 1989) were reviewed and the petrographic characteristics used to distinguish key kimberlite rock types and support macroscopic distinctions were found to be appropriate. No thin sections were available to review the petrography in detail.

The summary report and analysis of whole rock geochemistry data from 80 drill core samples spread across all three lobes from the pilot holes drilled adjacent to LDD holes (Stiefenhofer, 2007) was also reviewed. A distinct geochemical signature is apparent in plots of various trace element ratios (Zr/Ni; Nb/Y; TiO2/Y) for the South Lobe relative to the Centre and North Lobes; the Centre and North Lobes are compositionally similar. The geochemistry results were used to confirm observations from drill core logging and petrography, and to redefine the boundary between the Centre and South Lobes. The boundary between the South and Centre Lobes does, however, remains poorly constrained due to the spacing of drill holes from which the samples were obtained.

The approach to identifying different kimberlite units is deemed by MSC to be broadly appropriate and consistent with industry practice.

12.2 Drill core Logs

Drill core photographs for 44 of the 51 delineation (DDH) holes and 21 of the 23 pilot holes (PLT – drilled next to LDD holes) were reviewed by MSC and compared with digital lithology logs (assignment of intervals to kimberlite units) to evaluate the consistency and reliability of geological logging. No original paper logs were available for review. It was therefore not possible to check the original logging data and how this reconciles with kimberlite unit intervals recorded in the digital database. Nonetheless, the drill core photograph review provides a check on the overall consistency of the digital lithology logs as well as identification of potential errors therein. Based on observations of photos, logging data from 16 of the 74 drill cores were found to contain apparent minor errors in the form of an incorrectly applied logging code, incorrect down-hole distance of contacts, dilution estimation, and un-recognized intervals of distinct geology. In addition, three drill core logs (DDH041, DDH33, PLT 19) contained significant errors (>5 m down-hole difference) in the location of internal geology contacts and/or contacts between kimberlite and country rock, with potential implications for the overall pipe model volume. Nineteen holes were not photographed adequately to fully evaluate. While errors were detected during the review process, these were considered by MSC to be relatively minor and would not have a significant impact on the overall geology model and volume estimate for AK6.

In addition to the drill core photograph review, two drill cores were previously reviewed by MSA in 2010, and MSC reviewed an additional two drill holes in July 2013. These were found to be logged correctly and, in the case of the two drill holes reviewed by MSA, showed apparent consistency between the original logs and the digital logs (McGeorge et al., 2010). The positional accuracy and methodology of borehole surveys are commented on in McGeorge et al. (2010).

12.3 Internal Dilution

Estimates of the volume percent of wall-rock fragments (internal dilution) exceeding 0.5 to 1 cm in size were determined by line scan measurements over 0.3 and 0.5 m intervals from 67 of 74 drill cores at approximately 4 to 5 m spacing down hole. The methods and data used by De Beers to estimate average percentage dilution for each kimberlite unit in AK6 are considered by MSC to be appropriate and the results to be a reasonable representation of the overall levels of internal dilution present. The line scan method is not comprehensive, but appears to provide representative coverage of the AK6 deposit and is broadly consistent with industry best practice. Independent analysis by MSC of the line scan data yielded dilution estimates that were not materially different to those obtained by De Beers.

12.4 Geology Model

The original AK6 geology model was completed by Golder Associates Africa in 2007 (Opperman and van der Schyff, 2007). This model was further expanded by Farrow in 2007 by modelling a domain of expected high DMS yield (17+YIELD) within the South Lobe magmatic/pyroclastic kimberlite. The Anglo American Technical Division (MinRED) conducted a project review during the Front End Engineering Design (FEED) phase of the project in 2008, and recommended an update to the model (Rice et al., 2008). The 2008 3D geology model update by Tait and Maccelari (2008) incorporated these recommendations, and represented the most recent model version for AK6 prior to 2013. This model was reviewed in GEOVIA's GEMSTM software (GEMSTM) by MSC to verify that it represents a valid interpretation of the data available at the time.

The review of the 3D geology model did not identify any significant errors or concerns. MSC noted that the extent and geometry of the WM/PK(S) unit is very poorly constrained at present by drilling, in particular in the north-south direction, and thus may potentially be significantly larger than the interpretation indicated in the 2008 3D geology model. However, due to the relatively small size of this unit as well as the grade data and interpolation approach used for grade estimation (see Section 14.3), the mineral resource estimate of AK6 is not considered to be sensitive to potential variance in the size of WM/PK(S).

12.5 Bulk Density Data

The bulk density data used for estimation in 2008 were derived from sampling of drill cores from delineation drilling (2004 to 2006) and pilot holes drilled prior to the LDD drill holes (2005 to 2006). Bulk density (specific gravity) measurements were done on core samples using a water immersion method, by taking a 15 cm length of core and weighing it in air and in water and calculating moisture to derive wet and dry bulk densities (McGeorge et al., 2010). Details of the procedures followed are not available but the general approach used by De Beers is in line with industry best practise. MSC reviewed the dataset applied by De Beers in 2008 (Bush, 2008a), verified that bulk density samples were correctly coded according to the 2008 geology model solids, and further checked the data against original De Beers sample inventories for transcription errors. No significant data discrepancies were identified.

12.6 LDD Grade Data

Two large diameter drill (LDD) sampling programs were carried out in two phases from 2006 to 2007, during which a total of 30 holes comprising 8,635 m of 23 inch diameter drilling were completed. Samples comprising 12 m increments down hole were collected and processed from 24 of these LDD drill holes. The De Beers sample set used in the 2008 estimate (Bush, 2008a) was verified to conform to the 2008 geology model solids, and was checked against the original LDD sample results for transcription errors. This review identified several samples reported by Bush (2008a) that did not reflect the original LDD sample grades returned from processing and thus required correction before inclusion in the current mineral resource update. Grade data were further reviewed in 2016 following a reconciliation of extensive production against the 2013 resource estimate. The grade estimate was found to be locally over-estimated where the grade interpolation was strongly influenced by several statistical outlier LDD sample results. These outlier results were capped and grades were re-interpolated into the block model in which the resource estimate is hosted.

12.7 Production Data

12.7.1 Grade control data

The AK6 kimberlite has been mined for diamonds at Karowe Diamond Mine since April 2012. Detailed records of all kimberlite hauled are maintained by Karowe Mine. Individual truck haul tally sheets are maintained on a daily basis for each different aspect of kimberlite mining and stockpiling. These records include the truck type, time of each trip, departure location, tipping destination and the material type being transferred (rock type, kimberlite lobe and bench from which it was derived). These data are captured by Karowe staff into kimberlite depletion reconciliation workbooks, and survey volume calculations are used to verify the results obtained. These records provide a detailed breakdown of all stock movement on site and can be used with a high level of confidence to confirm the source material for plant production where the material was moved directly from the pit to the plant. While accurate records of stockpile material feed to the plant are maintained, kimberlite from different source locations is blended on the stockpiles. Thus, where stockpile material forms a significant component of the plant head feed, it is not possible to accurately reconcile production periods and diamond parcels with a source location in the pit. MSC did not undertake a comprehensive audit of the grade control database. However, several of the hard copy tally sheets were compared with the mineral resource depletion records to check for consistency and these were found to be accurate.

The survey equipment used to generate mine survey data include a Trimble S8 Total Station and a Fujiyama Hi Target V30 GNSS RTK system. Valid calibration certificates for both these systems were observed and the survey data generated are considered to be of acceptable quality.

12.7.2 Process data

The Karowe Mine plant process was briefly reviewed and QA / QC procedures in place are considered to be within or better than industry standards. Quality control checks are in place for all plant processes, including (but not limited to): weekly belt cut testing and calibration of weightometers; weekly tracer testing of DMS cut-point and recovery x-ray efficiency; daily particle size distribution granulometry studies at key points in the process stream; and regular data capture and monitoring of process-related information at hourly, daily and weekly levels as required.

12.7.3 Diamond data

Diamond data used for the updated mineral resource estimate documented in this report include recoveries by production batch sieved according to standard Diamond Trading Company (DTC) size classes from DTC1 to DTC23, with diamonds larger than 10.8 ct recorded separately. Size data generated on site were compared with size data from the Karowe Mine diamond facility in Gaborone, where diamond parcels are further sized and parcelled for sale, and a comprehensive audit of the individual weights of all +10.8 ct diamond was carried out. No significant discrepancies were noted. The diamond data used in this estimate are therefore considered to reliably reflect diamond production from the Karowe Mine up to the cut-off date for the estimate (21st October, 2013). Diamond production data in conjunction with sales data Table 6.1 have been used to modify and update the 2013 size distribution and value models (Section 14.5) for use in the PEA.

13 Mineral Processing and Metallurgical Testwork

During the Feasibility stage of the original mine project, it was already recognised that there were significant metallurgical risks in the ability of the grinding circuit to grind hard kimberlite below the weathered zone in order to effectively liberate diamonds and the ability of the DMS circuit to efficiently treat very high yield material expected from portions of the M/PK(S) geological domain in the South Lobe. After operations began in 2012, some exceptionally large, high value diamonds were recovered and the risk of damage to or breakage of these large diamonds was recognised as a further major risk to value generation. As described in the previous NI 43-101 Technical Report of February 2014, titled "Karowe diamond Mine Botswana NI43-101 Independent Technical Report and summarised below, testwork was commissioned to investigate technologies to mitigate these risks.

13.1 Comminution Testwork

Comminution testwork was undertaken in order to understand the effect of the harder kimberlite on the plant throughput and resultant product size. In consultation with Outotec, the suppliers of the mill, a number of mill simulations were carried out together with further comminution testing by SMC Testing (Pty) Ltd of actual mill feed and product samples taken when treating the hardest kimberlite. JKTech assessed and reported on the comminution results and indicated that the mill feed samples ranged from moderately hard to moderately soft with the large variability in competency of the mill feed affecting mill throughput.

13.2 XRT Testwork

In anticipation of expected increases in DMS yield, testing of X-Ray Transmission (XRT) technology to evaluate the applicability of XRT on different size fractions of Karowe kimberlite was successfully carried out in Germany in 2013 and 2016. Sensor based XRT bulk sorting has been implemented to recover diamonds from the sub-middles and coarse streams which reduces the load on the DMS subsequently treating a top size of 4mm.

13.3 Diamond Breakage Analysis

QTS Krystal Dinamika conducted a diamond breakage study on Karowe diamonds in 2013, 2015 and 2017. These studies have determined that although the Breakage Index was very low in comparison with similar operations producing large high value Type IIa diamonds, a significant amount of impact breakage and abrasion damage was observed, possibly caused by the pebble crusher and extended residence time in the mill. Further investigations continue to address damage and abrasion features.

13.4 Further Testwork

Other than the vendor XRT work mentioned above, no further comprehensive metallurgical testwork has been carried out since that assessed and described in the NI 43-101 of 2014. That testwork provided information for the plant modifications implemented in the Karowe Plant Phase 2 and Phase 3 upgrades.

14 Mineral Resource Estimates

This section summarises the data and methods used for updating the mineral resource estimate for the AK6 kimberlite. The previously reported mineral resource estimate (McGeorge et al., 2010) was based on the work of De Beers between 2002 and 2007 (see Sections 6, 9, 10 and 11), which culminated in a resource estimate in 2007 (Bush, 2007) that was reviewed and slightly modified by De Beers and Z-Star in 2008 (Tait and Maccelari, 2008; Bush, 2008a; Bush, 2008b; Bosma, 2008). The 2008 mineral resource estimate was derived by integrating the 2008 geology model with bulk density, diamond grade, and average diamond value data in a block model to provide local estimates of grade and bulk density and to determine total volumes, tonnes and diamond carats in AK6 (De Beers MRM, 2008). The updated resource estimate described in this report is presented in comparison with the 2008 estimate reported in McGeorge et al. (2010).

The process to update the mineral resource estimate from the previous estimate (McGeorge et al., 2010) can be summarised as follows:

- Following a careful review, MSC verified that the estimation approach taken by De Beers and Z-Star to generate local bulk density and grade estimates for the 2008 mineral resource model is appropriate and consistent with industry standards. The definition of variogram parameters and search constraints for the interpolation approach applied (ordinary kriging), while restricted in some cases due to data limitations, is considered to have been robust and the same approach was adopted by MSC for the current mineral resource update.
- The block model results reported by Bush (2008a) were successfully reproduced by MSC to ensure the correct geostatistical and model parameters were applied in the interpolation of the updated block model.
- The geology model was updated by MSC to reflect the findings from the data review by MSC and mapping by Karowe Mine geologists during recent mining activities.
- A block model with the same structure as that used for the 2008 estimate was constructed and updated by MSC in GEMSTM to reflect the updated geology model.
- The block model was populated with bulk density and grade values by interpolation of data from revised bulk density and LDD grade sample sets reflecting the new geology model.
- LDD sampling data were assessed in comparison with mine production data to derive a correction factor to convert modelled grade (based on 1.0 mm LDD diamond recoveries) to grade estimates recoverable by the Karowe Mine at its current operational parameters (1.25 mm bottom cut-off).
- Reconciliation of the grade estimates made in this way with production data in 2016 highlighted local anomalies where grade was over-estimated. This was investigated and was found to be related to a small number of highly anomalous grade results that were over-influencing proximal blocks. Outlier samples were identified and capped to remove this local over-estimation from the grade model. The block model was updated with this grade revision in 2016.
- LDD diamond results were investigated in detail and, in conjunction with the geology review, provide support for overall continuity in diamond size frequency distribution (SFD) with depth in each lobe of AK6.
- Diamond value estimates by size class, updated based on recent production data, were applied to recovery-corrected SFD models for production parcels from each of the lobes to derive average diamond values for each lobe. The diamond value estimates made in 2013 were further updated in 2017, based on the substantially larger production and sales database available, and have been updated accordingly in the block model.

The details of each component of the updated mineral resource estimate are described in the sections below.

14.1 Geology Modelling

The 2008 De Beers 3D geology model of AK6 was utilised as the basis for the updated geology model for the 2013 mineral resource estimate. MSC reviewed the methods and data used to generate previous versions of the AK6 geology model and was satisfied that they are appropriate and that the model is a reasonable representation of the available data. While geologically reasonable alternative models can be generated based on the available data, these are not considered to be significantly more likely than the models generated by De Beers and subsequently updated for the 2013 resource estimate. Furthermore, variations in volume estimates derived from realistic alternative models are within limits of uncertainty that MSC regards as appropriate for indicated and inferred mineral resources, respectively.

A further update to the geological model is in progress.

14.1.1 Geology model update and volume estimates

Adjusted drill logs from the review of core photographs (Section 12.2) were combined with information obtained from mapping in the open pit to update the geology model using. Surveyed pipe wall contacts derived from mapping in the open pit were provided by Karowe Mine geology staff for elevations between 1,008 and 941 mamsl. Mapped internal geology boundaries between the weathered and fresh kimberlite, and between kimberlite and breccia were provided for the elevation range 1,008 to 958 mamsl. The weathered nature of the kimberlite exposed in the open pit did not permit mapping of the boundaries between the North, Centre and South Lobes.

Key updates to the 2008 3D geology model of AK6 are described below and illustrated in Figure 14.1 to Figure 14.4:

- New kimberlite breccia zones (KBBX(S) and WKBBX(S)) were identified from open-pit mapping in the top 100 m of the west-southwest part of South Lobe (Figure 14.1). These zones were added to the BBX(S) and WBBX(S) domains, respectively, and were modelled to include drill core intersections with relatively high dilution (from DDH-13; DDH-09; PLT-021; PLT-04) previously included in the M/PK(S) domain.
- A new domain was modelled to encompass a large internal basalt block (INTBS(S)) that was identified by surface mapping in the upper 100 m of the west-southwest part of South Lobe (Figure 14.1).
- A new kimberlite breccia zone identified from surface mapping in the open-pit was modelled at ~50 m below surface in the southwest part of North Lobe and added to the BBX(N) domain (Figure 14.2).
- The North and Centre Lobe models were updated with a gap between them in the upper 80 m to reflect mapping results from the open pit (Figure 14.2).
- The outline of the North Lobe at surface was expanded based on surface mapping information, changing the surface area from 3,998 m^2 to 6,338 m^2 (Figure 14.3).
- The pipe margin in the uppermost 100 m on the west-southwest side of South Lobe was moved inwards based on mapping in the open pit resulting in a reduced surface area for this portion of the pipe (Figure 14.3).
- The weathering surface dividing “fresh” from “weathered” geology domains was adjusted in the 3D geology model to reflect only internal geology contacts between weathered and fresh kimberlite, and no longer includes pipe-wall contacts between wall-rock and weathered kimberlite. This change results in 20% increase in the volume of weathered kimberlite (Figure 14.4).

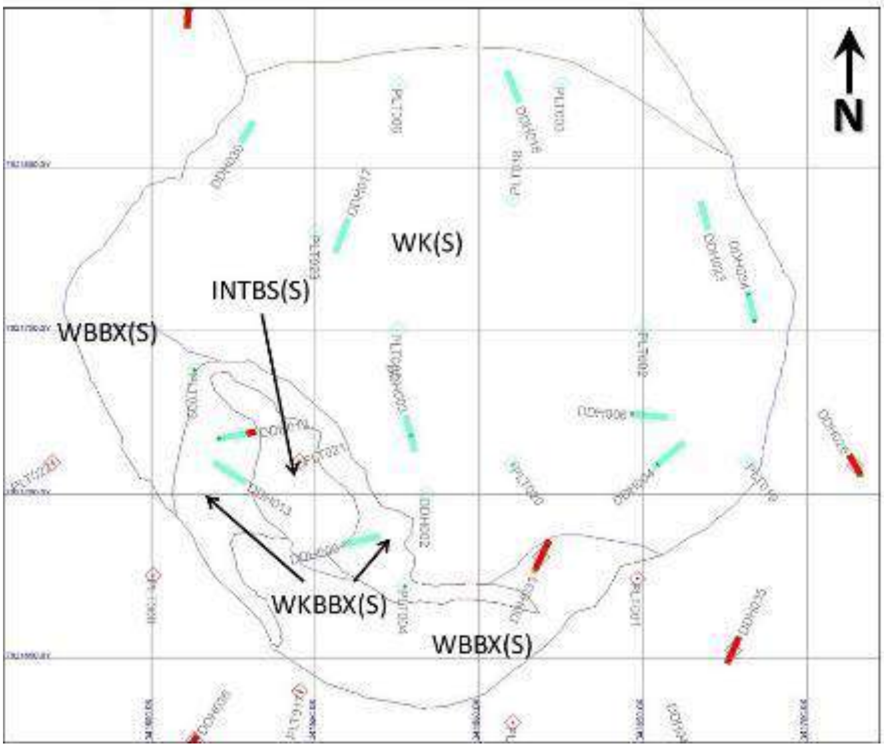


Figure 14.1 - Locations of new breccia zone and internal basalt domain in the South Lobe

Plan view of the South Lobe of the updated AK6 3D geology model showing the location of new modelled breccia zone WKBBX(S) and internal basalt domain INTBS(S) at 986 mamsl. Drill hole intersections of kimberlite (light blue) and basalt (red) for the horizontal plane corridor displayed are shown for reference.

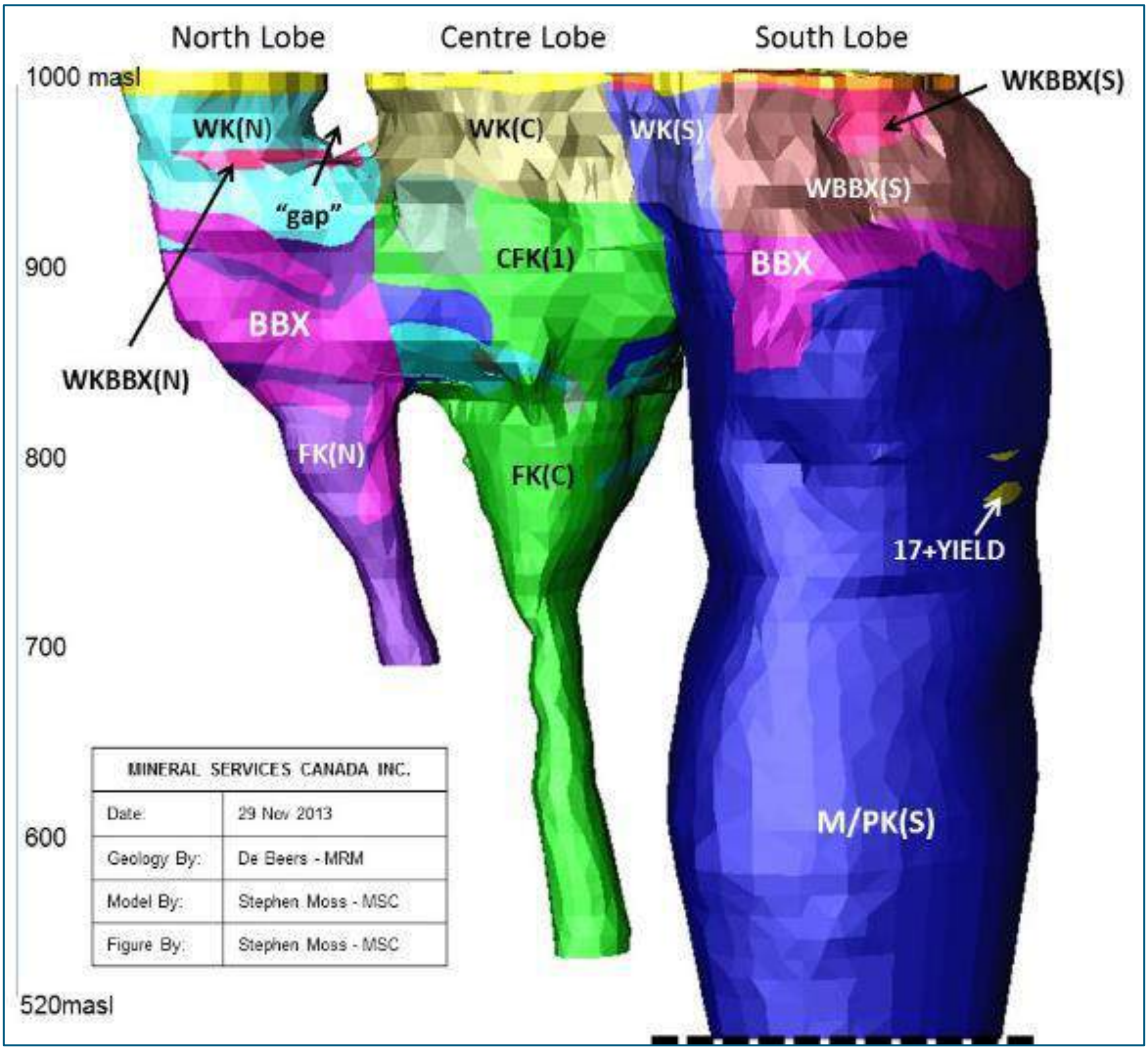


Figure 14.2 – Updates to the AK6 3D geology model in the North and South Lobes

View facing east showing the AK6 3D geology model from 1,000 mamsl to 520 mamsl. Arrows are shown indicating adjustments/additions to the model: the "gap" between the North Lobe and the Centre Lobe identified during mining; the WKBBX4(N) zone added to the BBX(N)); and the WKBBX(S) zone added to the WBBX(S) domain.

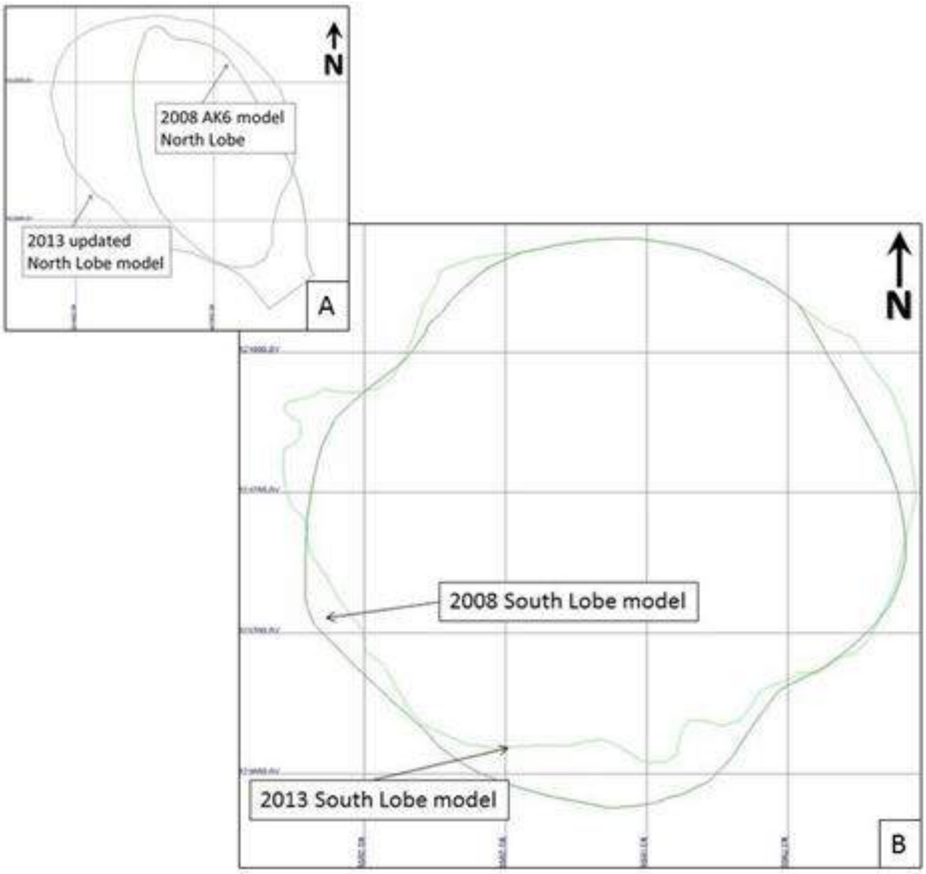


Figure 14.3 - Updated model geometry of the North and South Lobes

Plan views illustrating differences between the near-surface pipe outlines in the 2008 and 2013 geology models of the North (a) and South (b) Lobes. a) Plan view of North Lobe at 998 mamsl. b) Plan view of South Lobe at 1,005 mamsl. Gridline spacing on both maps is 50 m.

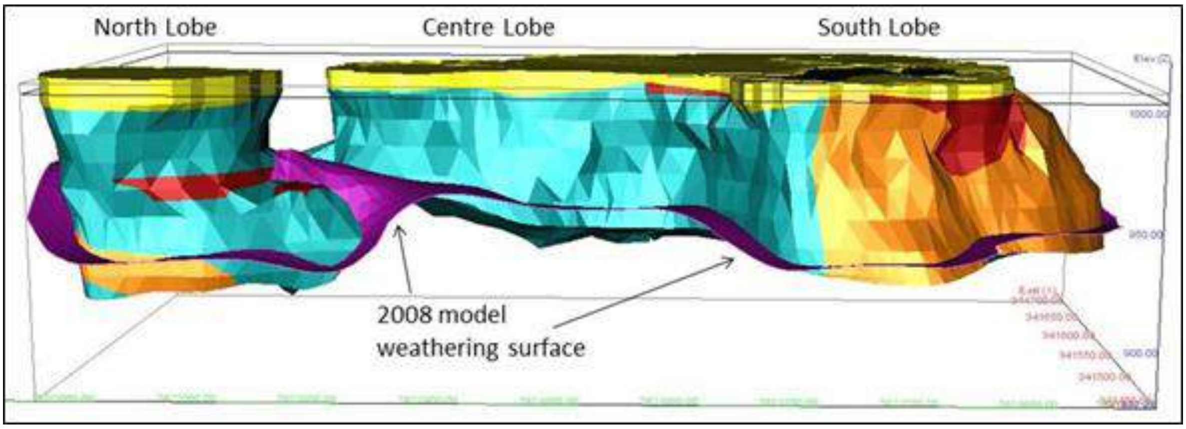


Figure 14.4 – Revisions to extents of weathered kimberlite domains

View of a portion of the AK6 geology model looking towards the east, illustrating the updated weathered kimberlite model solids in relation to the weathering surface from the 2008 model (purple). Depth changes of the bottom of the weathered kimberlite range from 10 to 30 m. Yellow = calcrete; pale blue = weathered kimberlite; orange = weathered basalt breccia; red = weathered kimberlite breccia.

Rock types identified in the AK6 kimberlite were grouped by De Beers into mappable kimberlite units defined based on their geological characteristics and interpreted grade potential (see Section 7.3). For the purposes of constructing the geological and mineral resource model, the kimberlite units occurring in each lobe were modelled in GEMSTM as geological wireframe solids, referred to in this report as domains (Figure 7.2;Table 7.2). The updated geology model that forms the basis for the AK6 updated mineral resource estimate comprises 13 kimberlite domains and 10 breccia / basalt domains. In cases where the unit is discontinuous (i.e. breccia units), the modelled domain comprises more than one discrete wireframe solid. Domain volumes are summarised and compared with the 2008 models in Table 14.1.

The updates to the 3D geology model are considered to be minor and represent refinement of the previous model based on the availability of new mapping data. Differences in the estimated domain volumes between the 2008 model and the current model are the primarily the result of adjustments to the extents of the North, Centre and South Lobes near surface, and the shifting of the weathering profile to a deeper depth. It is noted that there is a considerable degree of uncertainty in the size and distribution of large basalt blocks and breccia domains within each of the Lobes. There is also considerable uncertainty in the volume of dilution-rich kimberlite in the WM/PK(S) domain in the South Lobe. While the degree of uncertainty with regards to these factors is not considered to be sufficient to result in unacceptable degrees of uncertainty in the overall estimate of tonnes, carats and grades, it does indicate potential for significant variation from the model on a local scale that should be factored into future mine plans. No changes have been made to the geology model since the resource estimate reported by Lynn et al (2014).

Table 14.1 – Updated 3D model volumes

Geology domain	GEMS™ model volume (m^3)		Differences	
	2008	2013	Volume diff. (m^3)	% diff.
BBX(S)	76,550	82,679	6,129	8
CBBX(S)	59,426	42,346	-17,080	-29
CKIMB(S)	145,998	163,021	17,023	12
EM/PK(S)	1,307,806	1,307,595	-211	0
M/PK(S)	16,527,328	16,036,321	-491,007	-3
WBBX(S)	479,823	567,409	87,586	18
WK(S)	1,471,304	1,864,291	392,987	27
WM/PK(S)	188,478	188,478	0	0
17+YIELD	1,314,981	1,315,481	500	0
INTBS(S)[1]	na	51,523	na	na
South	**21,571,694**	**21,619,145**	**47,452**	**0**
BBX(C)[2]	587,183	268,126	-319,057	-54
CFK(C)	843,408	777,595	-65,813	-8
CKIMB(C)	75,240	88,341	13,101	17
FK(C)	1,471,651	1,433,860	-37,791	-3
WBBX(C)[2]	na	7,958	na	na
WK(C)	745,296	812,693	67,397	9
KBBX(C)[2]	na	70,457	na	na
Centre	**3,722,777**	**3,459,030**	**-263,747**	**-7**
KBBX(N)[2]	84,227	2,672	-81,555	-97
CKIMB(N)	28,936	69,798	40,862	141
FK(N)	549,707	421,040	-128,667	-23
WK(N)	172,538	296,310	123,772	72
BBX(N)[2]	na	282,381	na	na
WBBX(N)[2]	25,308	52,210	26,902	na
North	**860,716**	**1,124,411**	**263,695**	**31**
Sub-Total (North+Centre)	**4,583,494**	**4,583,442**	**-52**	**0**
Total	**26,155,187**	**26,202,587**	**47,400**	**0**

[1]Internal basalt block or "reef" identified by Karowe Mine geologists during mining has been included as a distinct geology domain

[2]The combined BBX, KBBX and WBBX of the North/Centre lobes from the 2008 model are split into separate domains corresponding to the North and Centre lobes, respectively, in the current model

Volume estimates per geological domain based on the current geology model (Lynn et al, 2014) compared to previous volume estimates (Tait and Maccelari, 2008).

14.1.2 Geological continuity

To determine whether the data obtained by processing of kimberlite at surface can be used as a basis for evaluating material from deeper parts of the corresponding lobes, the degree of geological continuity must be established within the key kimberlite units of AK6 with depth. Existing AK6 geology reports do not indicate any major geological discontinuity with depth, and grade variations within the individual lobes appear to be largely due to variable amounts of country-rock dilution (Stiefenhofer, 2007; Stiefenhofer and Hanekom, 2005). To confirm the degree of geological continuity, MSC reviewed surface exposure, drill core, petrography, geochemistry and dilution measurements. The key findings from this assessment are described below:

Surface and drill core observations

Limited examinations were made by MSC of kimberlite exposures in the open pit and in drill core during a site visit in July 2013. The geological continuity between the weathered (WK(S)) and fresh pyroclastic kimberlite (M/PK(S)) of the South Lobe was checked by examination of one drill core (DDH017), in which there was no evidence for a corresponding change in primary rock type. The observations did not highlight any major features or changes in the size and abundance of macroscopic constituents within the kimberlite that would support the presence of a major geological discontinuity at or close to the base of the weathered kimberlite. In contrast, the contact between M/PK(S) (South Lobe) and FK(C) (Centre Lobe) was clearly evident in this drill hole.

Drill core photograph observations

Drill core photographs for 44 of the 51 delineation (DDH) holes and 21 of the 23 pilot holes (PLT) were also reviewed and compared with digital lithology logs to evaluate the consistency and reliability of geological logging (see Section 12.2). During this review, macroscopic features of the drill core were examined to assess the apparent degree of geological continuity with depth for key rock types. Macroscopic observations of drill core photographs support the distinctions between M/PK(S) and FK(C), but do not indicate any significant differences between FK(C) and FK(N). The main kimberlite rock types within each lobe appear to be generally internally homogeneous with depth except for local variations in the size and abundance of country-rock xenoliths.

Internal dilution

Line-scan measurements of country-rock xenolith content (see Section 12.3) suggest very minor local variations in dilution within the main kimberlite rock type in South Lobe with depth (M/PK(S); Figure 14.5), with the exception of a small area intersected by two drill holes in the western half of the uppermost 100 m of the South Lobe. The amount of dilution present in FK(C) is on average approximately double that of the M/PK(S) in the South Lobe and shows a higher degree of horizontal and vertical variability (Figure 14.5). FK(N) has a similar dilution percentage to that observed in FK(C), and is also internally variable (Figure 14.5). With the exception of a possible decrease in dilution in FK(C) below 650 mamsl, the dilution data do not provide any evidence for significant geological changes with depth within the dominant kimberlite units making up AK6.

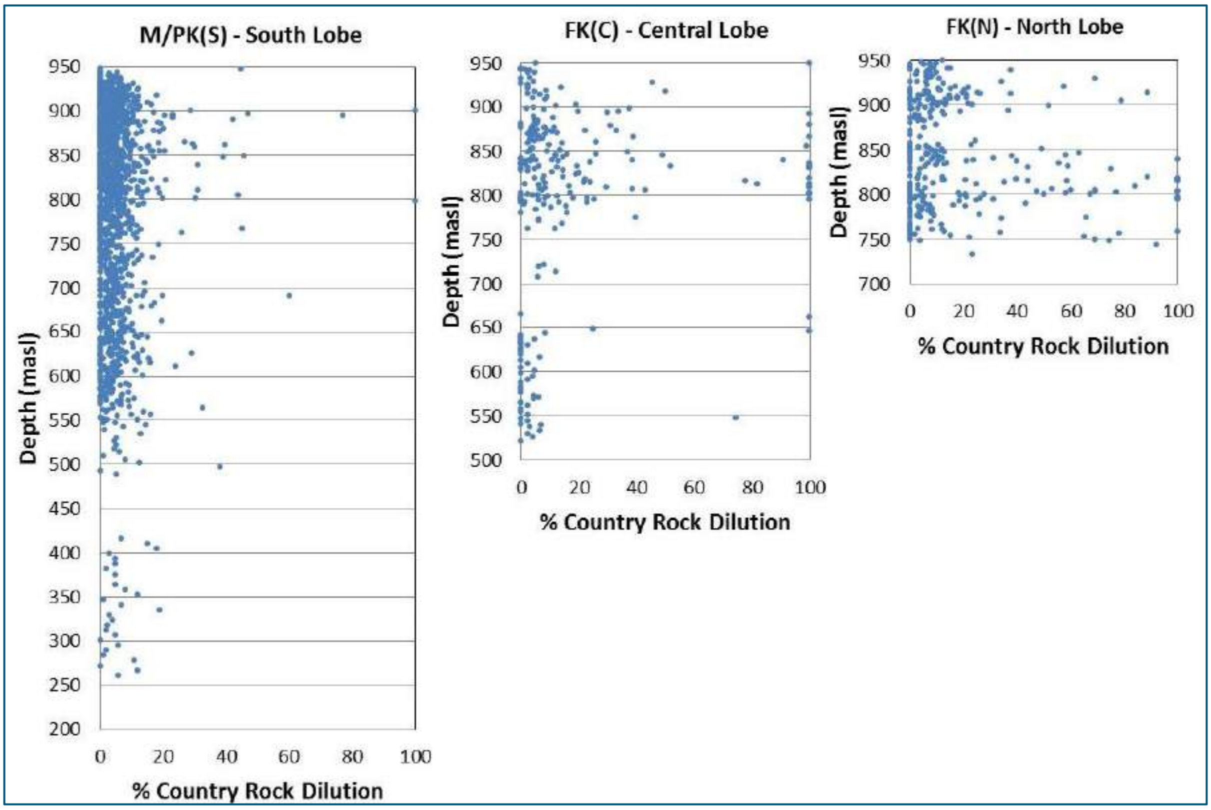


Figure 14.5 – Dilution with depth by lobe from drill core

Dilution with depth from line scan measurements of drill core for the main kimberlite rock types in each lobe: M/PK(S) – South Lobe; FK(C) – Centre Lobe; FK(N) – North Lobe.

The reviewed geological data support the distinctions made by De Beers geologists that between the different lobes of kimberlite at AK6: the North and Centre Lobes are broadly similar, but distinctly different from the South Lobe. These distinctions are apparent in a variety of observations, including the overall pipe shape, the sizes, shapes and abundances of mantle and country-rock components, groundmass mineralogy, the nature and degree of alteration products in the kimberlite, the bulk density and rock hardness. The observations made at site and in drill core photographs by MSC support the likelihood of geological continuity between the weathered surface material and underlying competent kimberlite in each of the three lobes. The only exception to this is the uppermost 10 m, wherein the significant calcretisation and weathering of the rock do not permit textural correlation with fresh rocks from depth in each lobe. Review of drill cores, drill core photos, dilution data and summaries of petrography data and geochemistry data suggest that, with the exception of local variations in the amount of country-rock dilution, the key kimberlite units identified at AK6 are internally homogeneous with depth.

14.1.3 Confidence level of geology model

The overall reliability of the geology model is considered by MSC to be moderate to high and appropriate for classification of volumes estimates from surface to the 600 mamsl elevation (~400 m below original surface) at a level of confidence appropriate for an Indicated mineral resource. This is based on the relatively high density of core and LDD drilling (Figure 14.6) as well as the thorough geological and geochemical approach taken to logging and definition of internal kimberlite units. Due to the significantly reduced density of drilling below 600 mamsl, the portion of AK6 from 600 mamsl to the base of the model at 260 mamsl is classified as an inferred mineral resource.

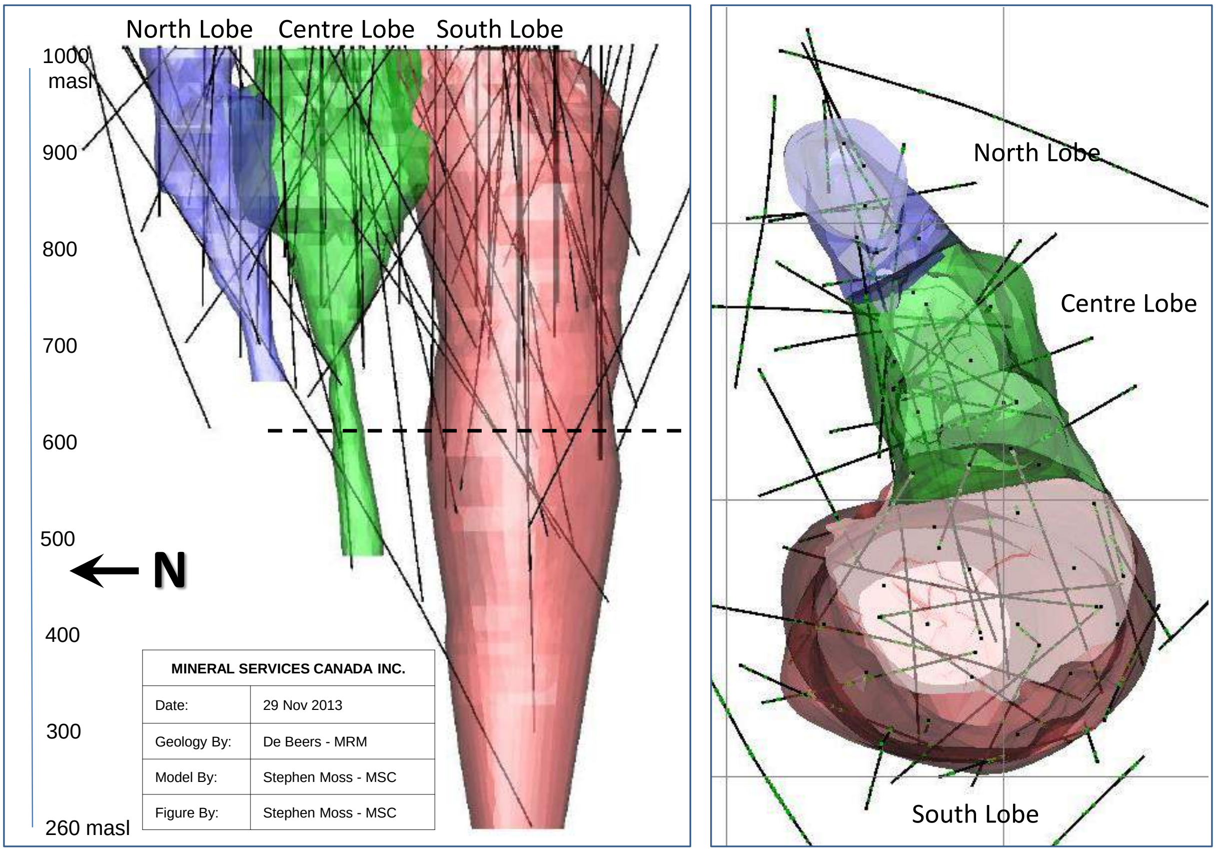


Figure 14.6 - Distribution of core holes drilled on the AK6 kimberlite

Plan (right) and section (left) views illustrating the distribution of core holes drilled on the AK6 kimberlite. These include vertical pilot holes drilled at effectively the same location as the LDD holes. The dashed line in the left panel indicates the base of the indicated resource (604 mamsl). Grid size on the right panel is 200 m.

14.1.4 Block model

A block modelling approach has been used for estimation of volumes, tonnes and grade for the AK6 kimberlite. The block model structure established for the 2008 estimate was used for the mineral resource update completed in 2013 (Lynn et al, 2013). It comprises 359,924 blocks with dimensions of 25 x 25 x 12 m, arranged in 68 rows (25 m wide), 79 columns (25 m wide) and 67 levels (12 m high). To accommodate the numerous domains present, a partial (percent) block modelling approach is required and was applied, for both the original 2008 estimate and the current update, using GEMS™ software. The block model folder structure was modified slightly to accommodate the minor revisions to the geology model, and was updated from the revised 3D geology solids with the percentage of each rock type within each block, calculated using the GEMS™ needling function with a horizontal needle orientation (3 x 3 needle density). Volumes for geology domains obtained from the block model were compared with the volumes of the 3D wire-frame solids and were found to be accurate to within 0.02%.

The block model was subsequently populated with bulk density and grade values based on the bulk density and grade samples as described in Sections 14.2 and 14.3 below. The grade and value estimates reported in 2013 were revised in 2016 and 2017, respectively; these updates have been incorporated into the existing block model to support this PEA.

14.2 Bulk Density and Tonnage

14.2.1 Bulk density data

The bulk density data used for estimation in 2008 derived from sampling of drill cores from delineation drilling (2004 to 2006) and pilot holes drilled prior to the LDD drill holes (2005 to 2006). The method used for bulk density determination is described in Section 12.5 above. During Phase I drilling, one sample was taken for every 7 m of drill core. During Phase II drilling, bulk density was measured every 3 m. The spatial distribution of bulk density samples is illustrated in Figure 14.7.

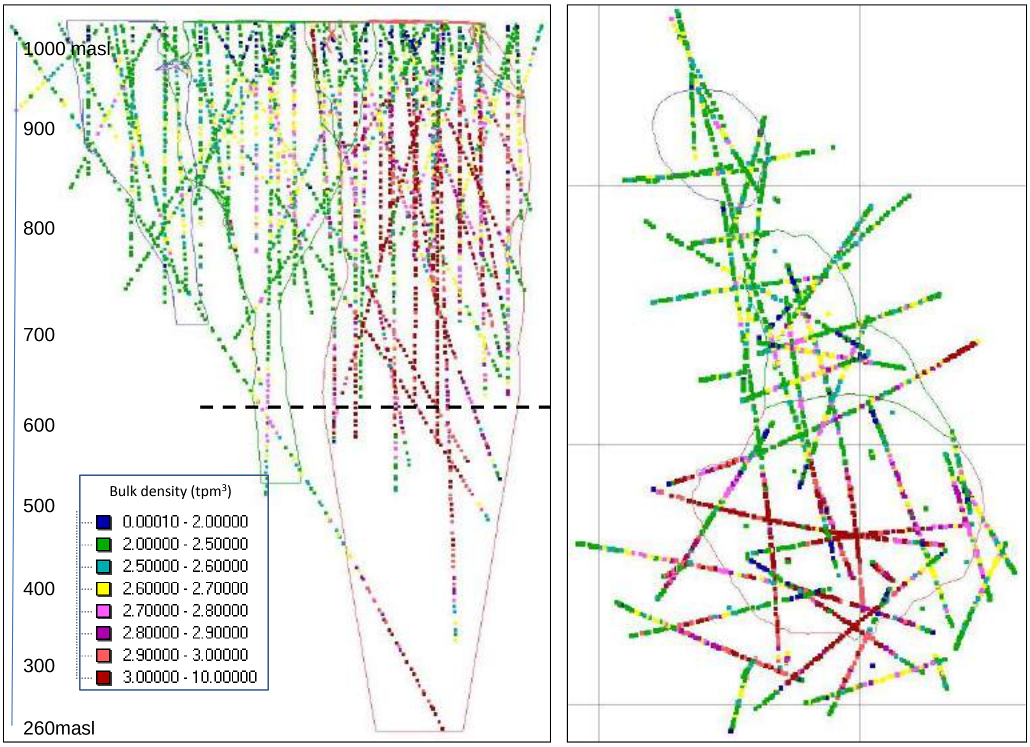


Figure 14.7 – Bulk density sample distribution

Distribution of bulk density (BD) samples throughout the AK6 kimberlite. Left panel: cross-section showing projected position of BD samples in relation to pipe outlines. Dashed line in cross section represents the base of the indicated mineral resource (600 mamsl). Right panel: Plan-view showing projected position of BD samples in relation to the pipe outline at 986 mamsl. Grid size on the right panel is 200 m.

Prior to estimation, the same sample set used by De Beers in 2008 (Bush, 2008a) was coded according to the updated geology model solids. As a result, a sub-set of samples used by De Beers in 2008 was excluded from the current bulk density estimate because the samples were not within the revised extents of the geology model. The final bulk density sample data used for the mineral resource update are summarized and compared to the 2008 averages (Bush, 2008a) in Table 14.2. The updates to the bulk density dataset resulted in only minor changes in the average sample bulk densities for each modelled geology domain.

Table 14.2 – Summary of bulk density data

Geology Domain	Bulk Density Group	2008 Density		2013 Density		% Diff
		n	Avg (tpm^3)	n	Avg (tpm^3)	
BBX(S)	Breccia	4	2.71	9	2.73	0.74
CBBX(S)	Breccia	4	2.23	3	2.19	-1.79
CKIMB(S)	S_Weathered	19	2.39	19	2.41	0.84
EM/PK(S)	S_Primary	123	2.76	122	2.77	0.36
M/PK(S)	S_Primary	1,040	2.86	976	2.88	0.81
WBBX(S)	S_Weathered	46	2.23	74	2.18	-2.36
WK(S)	S_Weathered	202	2.21	230	2.32	4.86
WM/PK(S)	S_Primary	43	2.56	44	2.56	-0.01
17+YIELD	S_Primary	135	3.00	132	3.01	0.27
INTBS(S)[1]	Breccia	na	na	9	2.36	na
South		**1,616**	**2.75**	**1,618**	**2.75**	**0.00**
BBX(C)[2]	Breccia	160	2.53	67	2.55	0.61
CFK(C)	C&N_Primary	171	2.61	156	2.61	-0.11
CKIMB(C)	C&N_Weathered	8	2.35	8	2.35	0.00
FK(C)	C&N_Primary	180	2.58	182	2.57	-0.44
WBBX(C)[2]	C&N_Weathered	0	na	0	na	na
WK(C)	C&N_Weathered	102	2.10	124	2.19	4.29
KBBX(C)[2]	C&N_Weathered	23	2.58	20	2.59	0.39
Centre		**644**	**2.50**	**557**	**2.49**	**-0.27**
CKIMB(N)	C&N_Weathered	7	2.29	8	2.26	-1.53
FK(N)	C&N_Primary	158	2.43	138	2.43	-0.05
WK(N)	C&N_Weathered	26	2.16	50	2.28	5.60
BBX(N)[3]	Breccia	0	na	86	2.53	na
WBBX(N)[2]	C&N_Weathered	3	2.63	9	2.42	-7.91
North		**194**	**2.39**	**291**	**2.43**	**1.59**
Total		**2,454**	**2.65**	**2,466**	**2.65**	**0.0008**

[1] Internal basalt block or "reef" identified by Karowe Mine geologists during mining

[2] Note: The BBX, KBBX and WBBX of the North/Central lobes of the 2008 model are split into separate domains corresponding to North and Centre lobes, respectively, in the current geology model

[3]KBBX(N) samples are included into BBX(N) totals

Summary of data used for bulk density estimation. The data are summarised by modelled geology domain (Lynn et al, 2014) and assigned to bulk density groups for interpolation purposes (see Section 14.2.2 below). n = number of samples; Avg = average; tpm^3 = tonnes per cubic meter.

Bulk density values for kimberlite are variable between lobes and between fresh and weathered kimberlite varieties in each lobe. The average bulk density of samples from domains of weathered kimberlite ranges from ~2.2 tonnes per cubic meter (tpm^3) in the Centre Lobe to ~2.5 tpm^3 in the South Lobe. The average bulk density of fresh kimberlite in the North and Centre Lobes ranges from ~2.4 to 2.6 tpm^3, whereas fresh kimberlite in the M/PK(S) domain that dominates the South Lobe is more dense (average of 2.9 tpm^3) and the 17+YIELD domain yields an average of 3.0 tpm^3.

14.2.2 Bulk density estimation approach

Two bulk density estimation approaches were used for the current estimate to reflect variations in the spatial distribution of bulk density samples. A spatially representative coverage of bulk density samples from drill core (Figure 14.7) allows for local estimation by interpolation (ordinary kriging) of sample bulk densities into the block model to a depth of 604 mamsl. The bulk density data were combined into sample groups (Table 14.2) based on geology (e.g. lobes; weathered vs. fresh; breccia vs. kimberlite) and verified with sample statistics. Model variograms derived by Bush (2008a; Table 14.3) are deemed appropriate for bulk density estimation and, together with appropriate neighbourhood ranges (Table 14.4) have been used as inputs for interpolation of bulk density into the block model by ordinary kriging. "Hard" boundaries were used between geology domains in different bulk density groups; i.e. bulk density data were not interpolated across boundaries between groups. Boundaries between different domains within a bulk density group were treated as "soft", i.e. bulk density values were interpolated across these boundaries. Ordinary kriging was mostly carried out in a single pass by using the neighbourhood searches shown in Table 14.4. A large area in the southwest of the South Lobe comprising BBX(S) and CBBX(S) was uninformed after the first pass interpolation. A second pass interpolation was applied to this area using larger search radii.

Table 14.3 - Variogram parameters for bulk density (BD) estimation.

BD Group	Nugget	Model	Sill	Range (m)			Model	Sill	Range (m)		
				X	Y	Z			X	Y	Z
South Primary	0.010	Sph	0.037	90	90	150					
South Weathered	0.025	Expo	0.056	61	61	61					
Centre&North Primary	0.011	Sph	0.024	173	173	173					
Centre&North Weathered	0.024	Sph	0.020	55	55	55					
Breccia	0.017	Sph	0.008	17	17	17	Sph	0.006	79	79	79

Sph = spherical, Expo = exponential.

Table 14.4 - Neighbourhood parameters for bulk density estimation.

BD Group	Minimum	Optimal	Search Radii (m)		
			X	Y	Z
South Primary	3	10	100	100	36
South Weathered	3	10	100	100	36
Centre&North Primary	3	10	120	120	48
Centre&North Weathered	3	10	100	100	36
Breccia	3	10	120	120	36

"Minimum" and "Optimal" refer to the number of samples used to interpolate a block.

At depths below 604 mamsl, drill core coverage is insufficient for local bulk density estimation of some kimberlite units. Based on geological continuity established by drilling, average interpolated block bulk densities from proximal benches or sample averages were applied to blocks uninformed by interpolation to generate semi-local bulk density estimates. A summary of the average bulk densities applied to uninformed blocks is shown in Table 14.5.

Table 14.5 - Summary of average (Avg) bulk densities (BD) applied to blocks uninformed by interpolation.

Block Model	Total blocks	Uninterpolated blocks	Avg BD (tpm^3)	Range of blocks for average
BBX(C)	193	3	2.53	850 to 826 mamsl
BBX(S)	78	3	2.73	None: sample average
CBBX(S)	31	2	2.19	None: sample average
M_PK(S)	3433	235	2.96	658 to 604 mamsl
WK(C)	238	3	2.19	None: sample average

Figure 14.8 shows the average block bulk density by mining level from the 2013 updated block model compared with average sample densities for the same levels. The final updated bulk density model is effectively unchanged from that of the original De Beers model (Bush 2008a; Figure 14.8 and Figure 14.9; Table 14.6) that formed the basis for the 2010 mineral resource estimate for AK6 (McGeorge et al., 2010). Minor differences between the current and the 2008 models are primarily due to the modifications made to the geology model (Section 14.1).

14.2.3 Confidence level of bulk density / tonnage model

The block bulk density estimates were combined with volumes estimates determined from the percent block model to generate an estimate of the tonnes of each kimberlite unit within each block. Due to the comprehensive sample coverage and careful statistical and geostatistical treatment of the data, the bulk density model between surface and the 600 mamsl elevation is considered to be of high confidence and suitable to support an indicate mineral resource classification. Due to the reduced sample density below 600 mamsl, bulk density for the lower portion of the pipe is less reliably constrained but estimates are considered to be at an appropriate level of confidence for an inferred mineral resource classification. In combination with confidence levels of volume estimates, as derived from the geology model (Section 14.1.3), the bulk density data support local estimates of kimberlite tonnes at an indicated level of confidence between surface and 600 mamsl and an inferred level of confidence between 600 mamsl and the base of the model at 260 mamsl.

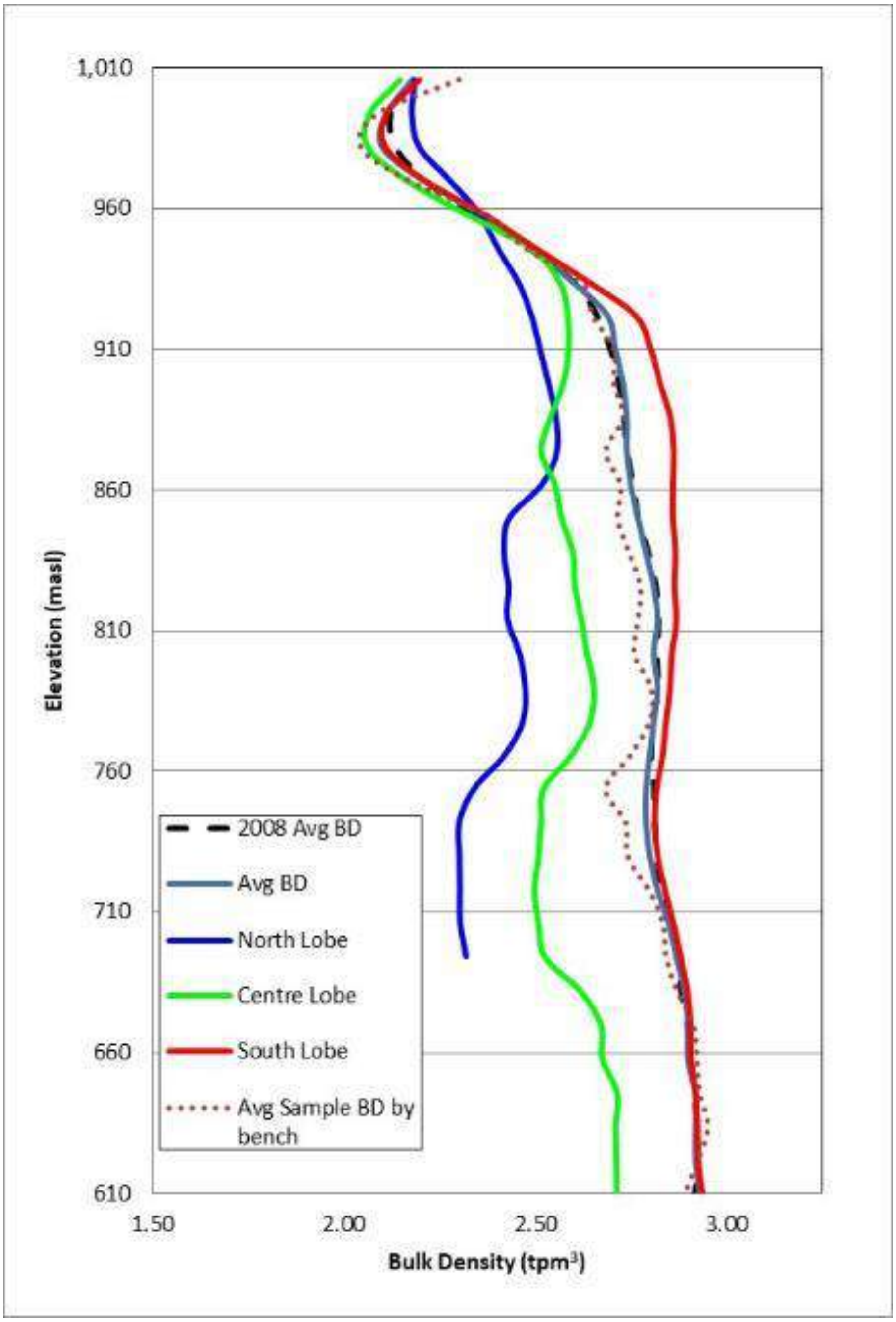


Figure 14.8 – AK6 block model bulk density profile with depth

AK6 bulk density profile with depth. Solid lines indicate the average block bulk density per 12 m bench for the entire AK6 body and for each of the individual lobes. The average block bulk density by level for the 2008 block model is shown for comparison. The dotted line indicates average sample values by 12 m bench.

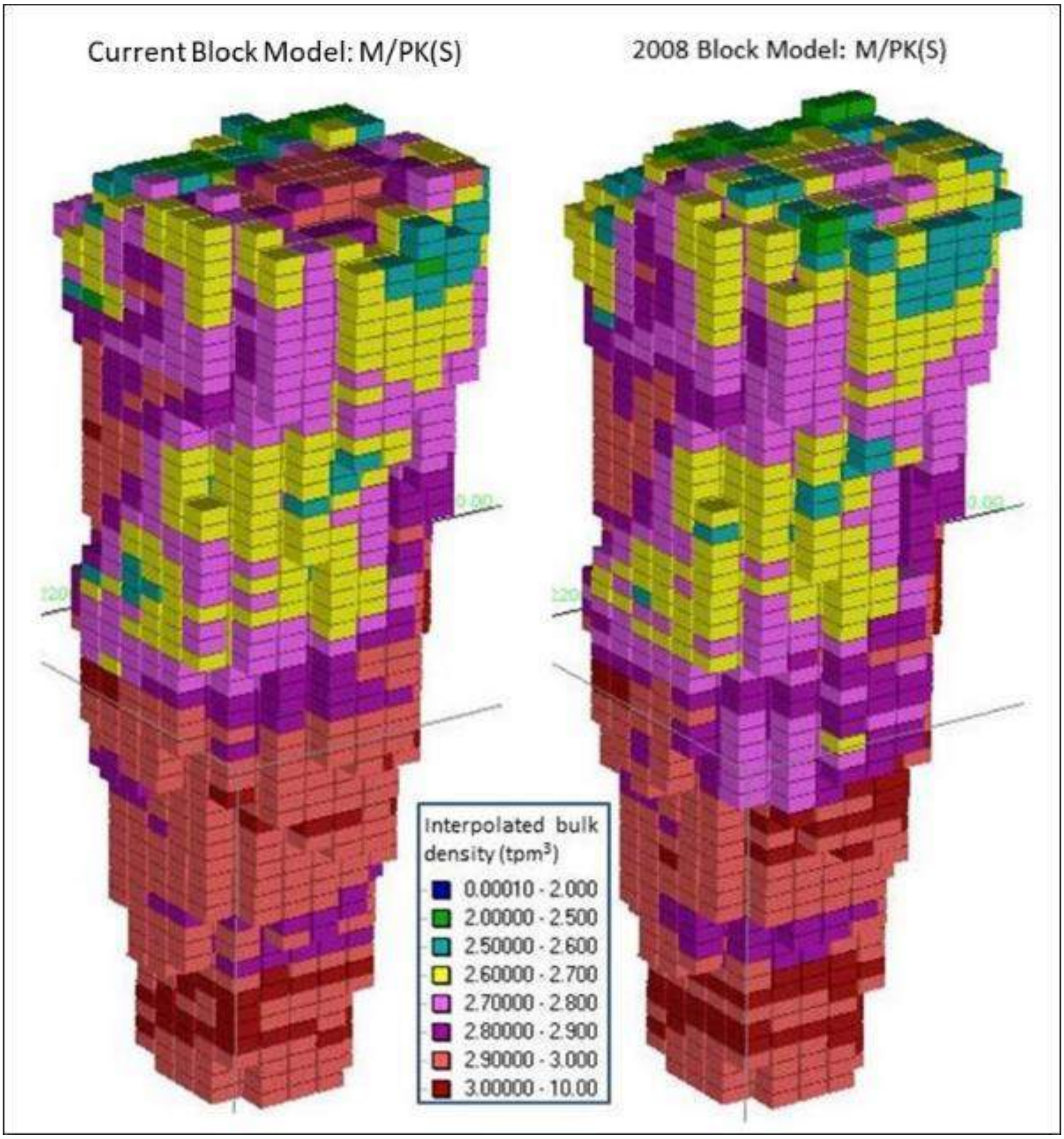


Figure 14.9 – Comparison of interpolated bulk densities in M/PK(S)

Inclined view facing NW showing comparison of interpolated bulk densities from 2008 block model and the current block model in M/PK(S). Blocks are 25 x 25 x 12 m in dimension. Blocks are coloured according to their interpolated bulk densities (in tonnes per cubic meter).

Table 14.6 - Comparison of average kriged bulk density by rock type

Geology domain	Sample BD (tpm^3)		Kriged (block) BD (tpm^3)	
	2008	2013	2008	2013
BBX(S)	2.71	2.73	2.60	2.79
CBBX(S)	2.23	2.19	2.33	2.32
CKIMB(S)	2.39	2.41	2.18	2.18
EM/PK(S)	2.76	2.77	2.77	2.84
M/PK(S)	2.86	2.88	2.85	2.88
WBBX(S)	2.23	2.18	2.21	2.28
WK(S)	2.21	2.32	2.23	2.32
WM/PK(S)	2.56	2.56	2.74	2.78
17+YIELD	3.00	3.01	2.97	2.95
INTBS(S)[1]	na	2.36	na	2.32
South	**2.75**	**2.75**	**2.76**	**2.76**
BBXC)[2]	2.53	2.55	2.55	2.56
CFK(C)	2.61	2.61	2.59	2.57
CKIMB(C)	2.35	2.35	2.15	2.14
FK(C)	2.58	2.57	2.59	2.56
WBBX(C)[2]	2.63	2.42	2.31	2.37
WK(C)	2.10	2.19	2.15	2.24
KBBX(C)[2]	2.58	2.59	2.59	2.59
Centre	**2.50**	**2.49**	**2.47**	**2.48**
CKIMB(N)	2.29	2.26	2.20	2.18
FK(N)	2.43	2.43	2.44	2.44
WK(N)	2.16	2.28	2.31	2.30
BBX(N)[3]	na	2.53	na	2.53
WBBX(N)[2]	2.63	2.42	na	2.34
North	**2.39**	**2.43**	**2.47**	**2.41**
Total	**2.65**	**2.65**	**2.76**	**2.76**

[1] Internal basalt block or "reef" identified by Karowe Mine geologists during mining has been included as a distinct geology domain in the current model

[2] The BBX, KBBX and WBBX of the North/Central lobes from the 2008 model are split into separate domains corresponding to the North and Centre lobes, respectively, in the current geology model

[3] KBBX(N) samples are included into BBX(N) totals

Sample and kriged block averages of bulk density (BD) by geology domain (as reported in Lynn et al, 2014). Sample and kriged block averages from 2008 block model by De Beers (Bush, 2008a) are shown for comparison. tpm^3 = tonnes per cubic meter.

14.3 Diamond Grade

14.3.1 LDD sample data

Two large diameter drill (LDD) sampling programs were carried out in two phases from 2006 to 2007, during which a total of 30 holes comprising 8,635 m of 23” diameter drilling were completed. Samples comprising 12 m increments down hole were collected and processed from 24 of these LDD drill holes. Holes were drilled vertically and are well-distributed across the pipe (Figure 14.10). Sample volumes were measured by caliper survey of all holes. These were used in conjunction with the carats of diamonds recovered from each sample to calculate sample grades in carats per cubic meter (cpm^3).

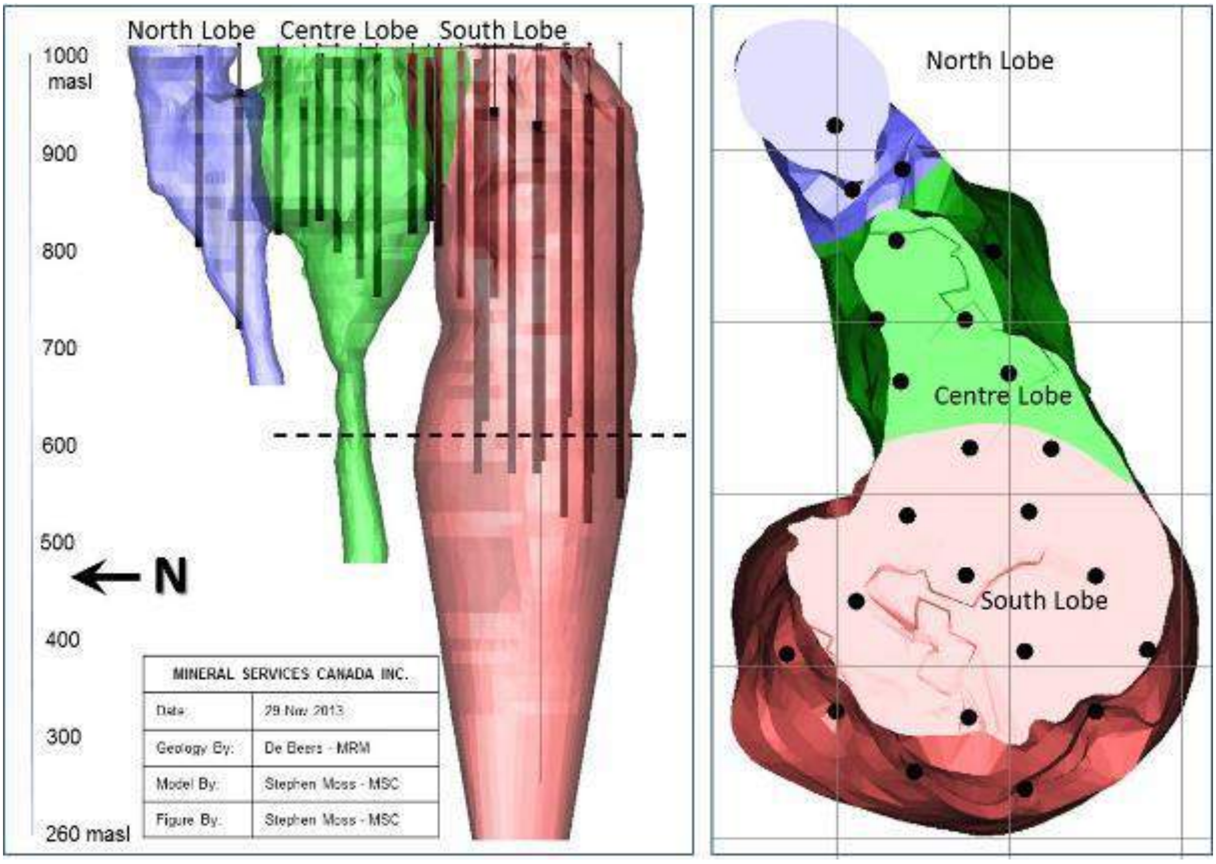


Figure 14.10 – LDD drill hole distribution

Distribution of AK6 LDD drill holes from which bulk samples were collected. Left panel: Cross section view facing east showing depth extents of LDD sample coverage (thick black trace) in relation to projected pipe lobe models (thin black trace represents unsampled portion of LDD drill hole). Dashed line shows the base of the indicated mineral resource (600 mamsl). Right Panel: Plan view showing collar locations of sampled LDD drill holes. Grid size on the right panel is 100 m.

The De Beers sample set from 2008 (Bush, 2008a) was coded to the geology model solids. As for the bulk density samples, some of the recoded LDD samples are completely or predominantly (>60%) outside of the remodelled pipe, and were thus excluded from the estimation dataset, along with all recoveries not directly attributable to samples (e.g. spillage and process purge during processing). A few samples not used by Bush (2008a) because they were outside of the model were included in the revised grade estimate because of local changes to the geology model. These data formed the basis of the grade estimates reported in Lynn et al (2014).

The 2013 grade estimates were modified in 2016 following an assessment of LDD stone frequencies (st/m^3), average stone sizes (MSS) and grade (cpm^3) data to identify outlier samples. Anomalous values (greater than 3 standard deviations from the mean) were corrected as follows:

- In the South Lobe, 4 samples (DCD781, DEC758, DCD756 and DCE227) displayed anomalous stone frequency values. Grade values for these samples were capped by reducing the sample stone frequencies to the highest st/m^3 value from South Lobe which was less than three standard deviations from the mean (30.8 st/m^3), and applying the MSS of the sample to calculate a capped grade (cpm^3). A single sample (DCD722) with an anomalous grade value (4.93 cpm^3) was truncated to a grade matching the highest grade determined by the capping process described above for samples with anomalous stone frequencies (3.26 cpm^3).
- In the Centre and North Lobes, 1 sample (DCE146) displayed an anomalous mean stone size and anomalous grade due to the recovery of a 13.37 ct diamond. The single large diamond was removed from this sample and

its grade was recalculated at 1.08 cpm^3. Two additional samples (DCD664 and DCD710) with anomalous grade values were truncated to the highest recovered grade less than 3 standard deviations from the mean (2.31 cpm^3).

The LDD sample grade data used to generate the 2016 updated grade model are summarised by modelled geology domain in Table 14.7 and LDD diamonds recoveries by lobe and DTC size class are provided in Table 14.8. Comparison with the original LDD sample data set used in support of the 2008 resource estimate (McGeorge et al., 2010; Bush, 2008a) reveals minor differences in average sample grade per geology domain. These can be readily explained by updates to the geology model made in 2013 and/or corrections of transcription errors from the original LDD sample results in previous estimates (see Section 12.6).

Table 14.7 - Summary by geology domain of LDD sample grade data used for grade estimation in 2016.

Geology Domain	Grade Group	Grade Samples	
		Number	Avg Grade (cpm^3)
BBX(S)	Breccia	1	0.05
CBBX(S)	Breccia	0	na
CKIMB(S)	S_Primary	2	0.01
EM/PK(S)	S_Primary	43	1.08
M/PK(S)	S_Primary	213	0.52
WBBX(S)	Breccia	12	0.22
WK(S)	S_Primary	61	0.50
WM/PK(S)	S_Primary	13	0.60
17+YIELD	S_Primary	38	0.35
INTBS(S)[1]	na	0	na
South		**383**	**0.55**
BBX(C)[2]	Breccia	7	0.35
CFK(C)	C&N_Primary	40	0.76
CKIMB(C)	C&N_Primary	0	na
FK(C)	C&N_Primary	26	0.63
WBBX(C)[2]	Breccia	0	na
WK(C)	C&N_Primary	26	0.42
KBBX(C)[2]	Breccia	3	0.59
Centre		**102**	**0.61**
CKIMB(N)	C&N_Primary	0	na
FK(N)	C&N_Primary	25	0.49
WK(N)	C&N_Primary	8	1.23
BBX(N)[3]	Breccia	10	0.28
WBBX(N)[2]	Breccia	2	2.28
North		**45**	**0.65**
Total		**530**	**0.57**

[1] Internal basalt block or "reef" identified by Karowe Mine geologists during mining.

[2] The BBX, KBBX and WBBX of the North/Centre lobes from the 2008 model are split into separate domains corresponding to the North and Centre lobes, respectively, in the 2013 geology model.

[3] KBBX(N) samples are included into BBX(N) totals.
Avg = average. cpm^3 = carats per cubic meter

Table 14.8 - Total LDD diamond recoveries by DTC size class grouped by lobe

Size	North Lobe		Centre Lobe		South Lobe	
	Ct	St	Ct	St	Ct	St
DTC-1	0.05	0	0.18	0	1.56	0
DTC1	1.42	105	5.72	408	26.23	1876
DTC2	1.78	78	11.13	519	38.65	1802
DTC3	7.74	224	26.17	749	110.93	3179
DTC5	8.32	117	19.57	269	91.8	1266
DTC6	9.72	110	19.08	214	75.67	856
DTC7	10.49	87	23.67	192	84.68	695
DTC9	13.44	66	33.93	165	98.79	480
DTC11	13.1	35	23.65	66	65.36	182
DTC12	7.78	13	9.83	18	38.31	72
DTC13	8.14	11	22.6	27	58.85	70
DTC15	2.35	3	6.06	6	13.52	12
DTC17	7.76	6	10.61	7	23.25	16
DTC19	2.27	1	13.47	6	46.59	21
DTC21	0	0	0	0	27.02	6
DTC23	0	0	13.37	1	7.98	2
Totals	**94.36**	**856**	**239.04**	**2,647**	**809.19**	**10,535**
Volume (m^3)	**145**		**369**		**1309**	
cpm^3	**0.652**		**0.648**		**0.618**	

cpm^3 = carats per cubic meter

14.3.2 Grade estimation approach

Two grade estimation approaches were used for the current estimate, reflecting variations in the spatial distribution of LDD samples.

A local grade estimation approach has been applied from surface to 604 mamsl where a spatially representative coverage of LDD sampling allows for interpolation (ordinary kriging) of the + 1.0 mm sample grades into the block model. The grade data were combined into groups (Table 14.7) on the basis of geology (e.g. lobes; breccia vs. kimberlite) and grade sample statistics. In contrast to the bulk density analysis, grade groups did not distinguish equivalent weathered and fresh kimberlite types (i.e. these were included in the same groups). The variogram parameters determined by Bush (2008a;Table 14.9) were found to be appropriate despite the minor changes to the geology model and were used, together with the kriging neighbourhood parameters indicated in Table 14.10, as inputs for local grade estimation by ordinary kriging. There are insufficient data from the breccia units for variography (Bush, 2008a). Thus, the variograms for the associated grade group for primary kimberlite were used for the breccia units from each lobe. As for bulk density, boundaries between geology domains belonging to different grade groups were treated as “hard” in the interpolation process (sample data not interpolated across these boundaries). Boundaries between different domains within a grade group were treated as “soft”, i.e. grade values were interpolated across these boundaries. Two kriging passes were carried out for each group. The second pass comprised a larger search neighbourhood (Table 14.10) and was used to populate blocks uninformed from the first pass. For the South Lobe, the larger neighbourhood was used for the Breccia group only.

LDD sample coverage does not extend significantly below 600 mamsl and, where reliable local grade estimation is not possible, lower confidence global grade estimates have been applied. In these instances the average block grades from directly overlying or adjacent equivalent rock types were applied to the underlying areas to produce global grade estimates. A summary of the average grades applied to uninformed blocks is shown in Table 14.11.

Table 14.9 - Variogram parameters for local grade estimation

Grade Group	Nugget	Model	Sill	Range (m)		
				X	Y	Z
South Primary	0.120	Spherical	0.175	115	115	83
Centre&North Primary	0.172	Spherical	0.133	90	90	77

Table 14.10 - Neighbourhood parameters for local grade estimation.

Grade Group	Minimum	Optimal	Search Radii (m)		
			X	Y	Z
South Primary (first-pass)	3	10	100	100	48
South Primary (second-pass)	3	10	150	150	96
C&N Primary (first-pass)	3	10	100	100	60
C&N Primary (second-pass)	3	10	150	150	108

"Minimum" and "Optimal" refer to the number of samples used to interpolate a block

Table 14.11 - Summary of average grade values (carats per cubic meter) applied to blocks uninformed by interpolation

Block Model	Total blocks	Uninterpolated blocks	Average grade (cpm^3)	Range of blocks for local average
M_PK(S)	3433	1100	0.56	658 to 610 mamsl
EM_PK(S)	428	89	0.57	610 to 574 mamsl
FK(C)	590	78	0.51	730 to 706 mamsl

Figure 14.11 shows the average block grade by mining level from the 2016 update to the block model compared with average sample grades for the same levels. The final updated +1.0 mm grade model is very similar to that of the original De Beers model (Bush 2008a; Figure 14.12; Table 14.12) that formed the basis for the 2010 mineral resource estimate for AK6 (McGeorge et al., 2010). Minor differences between the current and the 2008 models are primarily due to the modifications made to the geology model (Section 14.1.1), slight changes made to the applied LDD sample dataset (Section 12.6) and the sample grade capping exercise described in Section 14.3.1.

Estimated block grades in carats per cubic meter are combined with interpolated bulk density to calculate grades expressed as carats per hundred tonne (cpht) for each block.

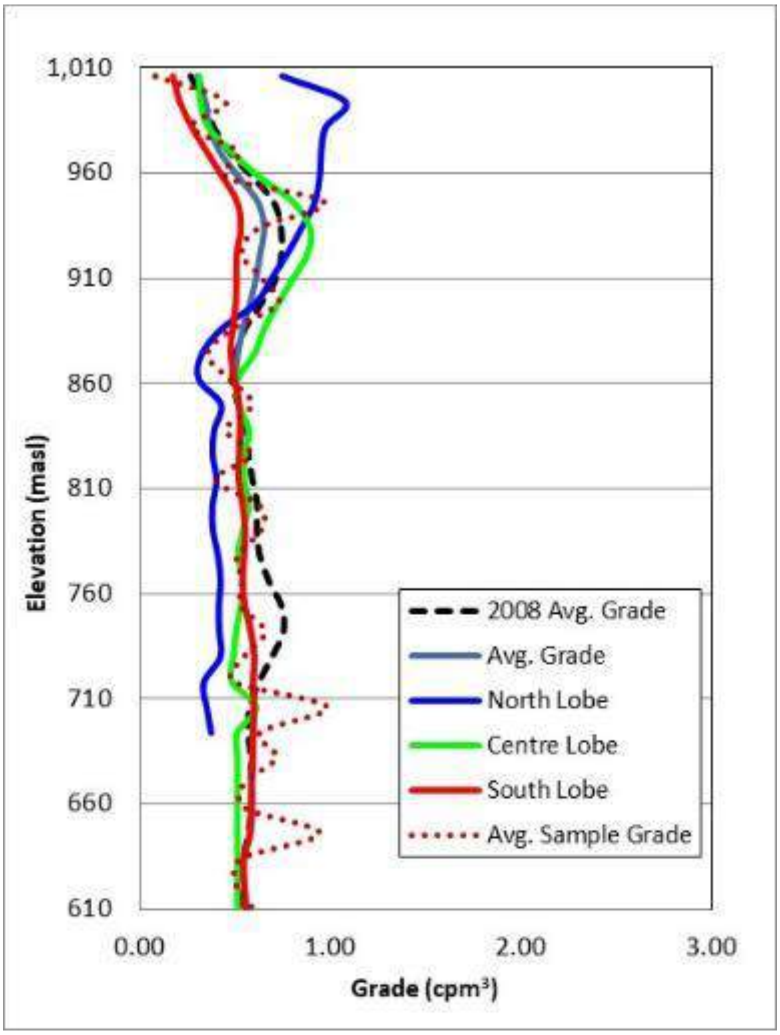


Figure 14.11 – AK6 block model grade profile with depth

AK6 grade profile (as revised in 2016) with depth. Solid lines indicate the average block grade per 12 m level for the combined AK6 body and for each of the individual lobes. The average block grade by level for the 2008 block model is shown for comparison. The dotted line indicates average sample values by bench

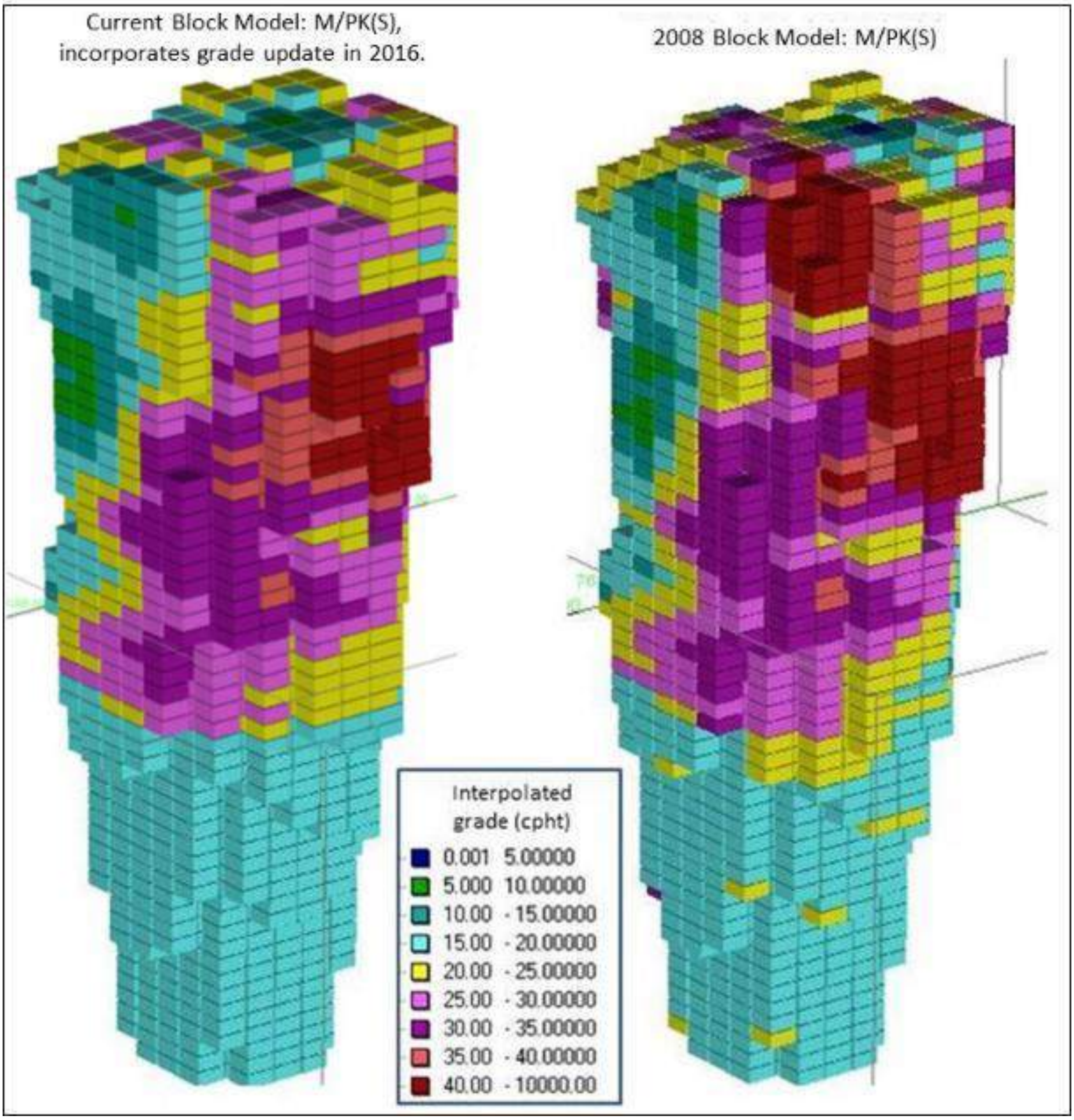


Figure 14.12 – Comparison of interpolated grades in M/PK(S)

Inclined view facing NW comparing interpolated grades for the M/PK(S) domain from the 2008 and 2013 block models. Grade estimates shown in the 2013 block model reflect updates made in 2016. Blocks are 25 x 25 x 12m in dimension. Blocks are coloured according to their interpolated grades (in carats per hundred tonnes)

Table 14.12 - Comparison of average kriged grade by rock type

Lobe	Rock Type	Grade Samples (cpm^3)		Avg Kriged Block Grade (cpm^3)	
		2008	2016	2008	2016
South	BBX(S)	0.12	0.05	0.14	0.21
	CBBX(S)	0.00	na	0.14	0.17
	CKIMB(S)	0.03	0.01	0.19	0.23
	EM/PK(S)	1.20	1.08	1.00	0.85
	M/PK(S)	0.56	0.52	0.59	0.57
	WBBX(S)	0.13	0.22	0.15	0.20
	WK(S)	0.50	0.50	0.42	0.49
	WM/PK(S)	0.85	0.60	0.58	0.55
	17+YIELD	0.35	0.35	0.42	0.43
	INTBS(S)[1]	na	na	na	0.00
Centre	BBX (N/C to C)[2]	0.45	0.35	0.45	0.48
	CFK(C)	0.86	0.76	0.79	0.74
	CKIMB(C)	na	na	0.30	0.34
	FK(C)	0.58	0.63	0.64	0.65
	WBBX(N/C to C)[2]	na	na	0.86	1.16
	WK(C)	0.36	0.42	0.52	0.56
	KBBX(C)[2]	0.46	0.59	0.43	0.43
North	CKIMB(N)	na	na	0.69	0.77
	FK(N)	0.61	0.49	0.64	0.59
	WK(N)	1.74	1.23	1.06	0.88
	BBX(N/C to N)[3]	na	0.28	na	0.52
	WBBX(N/C to N)[2]	na	2.28	na	1.06

[1] Internal basalt "raft" or "reef" identified by Karowe Mine geologists during mining.

[2] Note: The BBX, KBBX and WBBX of the North/Centre lobes from the 2008 model are split into separate domains corresponding to the North and Centre lobes, respectively, in the 2013 geology model.

[3] KBBX(N) samples are included into BBX(N) totals.

Comparison of sample and kriged (block) grades by rock type in AK6. Average sample and kriged grades from the 2008 block model by De Beers (Bush, 2008a) are also shown for comparison. cpm^3 = carats per cubic meter

14.3.3 Adjustment for recoverable grade

The grade estimates presented in the section above have been made on the basis of grade data from LDD samples processed with a 10 tonne per hour mobile DMS plant at a 1.0 mm bottom cut off. A recovery correction is required in order to convert these grades to an estimate of grade recoverable with the current Karowe plant (1.25 mm bottom cut-off).

In order to provide a basis for determining an appropriate correction factor, the size frequency distribution (SFD) of the LDD diamond recoveries was compared to that of production data from the Karowe Plant at its current configuration with a lower cut off of 1.25 mm (see Figure 14.13 below). This highlights a significantly lower recovery of diamonds in the smaller size ranges (less than ~ 0.2 ct) during production relative to LDD sample results for equivalent material, reflecting differences in liberation and recovery at the finer end of the diamond size range. Adjusting the LDD diamond SFD for each lobe to match that obtained during mine production from equivalent material results in an overall reduction of LDD grades by between 18% and 31%. This implies that a grade correction of approximately this magnitude is required to adjust the 1.0 mm grade estimate to an estimate of recoverable grade by the current Karowe plant. For the purpose of the current mineral resource estimate, an average recovery correction of 25% has been assumed.

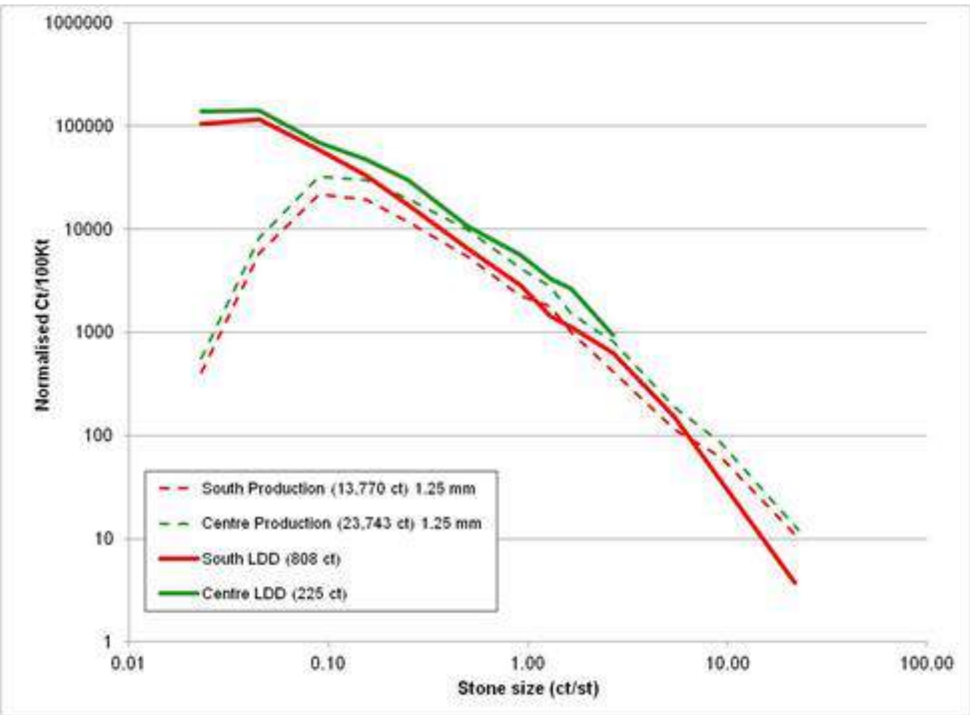


Figure 14.13 – Grade size curves for AK6 production and LDD diamond parcels

Grade-size curves comparing diamond production at 1.25 mm lower cut off to LDD sample data (1.0 mm lower cut off) for the Centre and South Lobes. See Section 14.4 below for explanation of datasets used for this comparison

While it is not possible to accurately verify the recovery correction factor based on available data, reconciliation of production data against the 2008 1.0 mm grade model provides support for the factor derived by SFD analysis. Two approaches were used for the reconciliation:

a) Reconciliation of selected short production periods during which the material processed was derived from one excavation location, with no significant addition of stockpile material and no concurrent stockpiling of (potentially lower grade) material from the excavation location. Five production periods were found to fulfil these criteria, amounting to 56 days of production, during which 385,000 wet tonnes were processed to produce 73,000 ct. Modelled solids were generated for each of these production batches based on surveys of the mine surface (provided by the Karowe Mine survey department) at the beginning and end of each period. These solids were intersected with the block model to generate predicted 1.0 mm grade estimates for each production period. Comparison of the estimated grade from the block model with the actual grade recovered from each production period (Figure 14.14) shows that in all cases the block model grade overestimates recoverable grade and, on average, the 1.0 mm block model grade is 21% higher than that achieved during production.

b) As further verification, a larger scale reconciliation of the 1.0 mm grade model with production and stockpile data was carried out (Table 14.13). Production data spanning the period November 2012 to October 2013 (consistent plant recovery parameter of 1.25 mm bottom cut off) were collated and were compared with estimates of carats contained within the portion of the deposit mined during this period (as determined from survey data provided by the Karowe Mine survey department) based on the block model (see Table 14.13 below). The grade of material on the stockpiles is not reliably known so a full reconciliation is not possible. However, if no correction factor is applied to the block model grade estimates, production records would imply an average stockpile grade (determined by subtracting actual tonnes processed and carats produced from the total estimate tonnes and carats mined during the period in question) of ~16 cpht. This is considered highly unlikely because of current grade control and kimberlite handling procedures at the Karowe mine (see Section 14.4.1 below) that involve a preferential allocation and / or retention of diluted lower-grade kimberlite to the stockpile, implying that production grades should be significantly higher than the average grade of material remaining on stockpiles. If a 25% correction is applied to the total 1.0 mm carats estimated to have been removed during the relevant production period, the estimated grade of the material stockpiled during this period would be on the order of 4 cpht (Table 14.13). This is considered to be a realistic average estimate of the recoverable grade of the material stockpiled, and provides further support for a downward correction to the 1.0 mm model grades of approximately 25%, to produce "recoverable" grade estimates for the Karowe Mine process plant operating in its current configuration.

It is important to note that the recovery correction factor determined for the 2013 update of the mineral resource estimate is appropriate for the current Karowe plant configuration as well as the physical characteristics of the kimberlite processed between November 2012 and October 2013. Significant changes to the plant configuration have been undertaken and the recovery correlation factor is still considered valid.

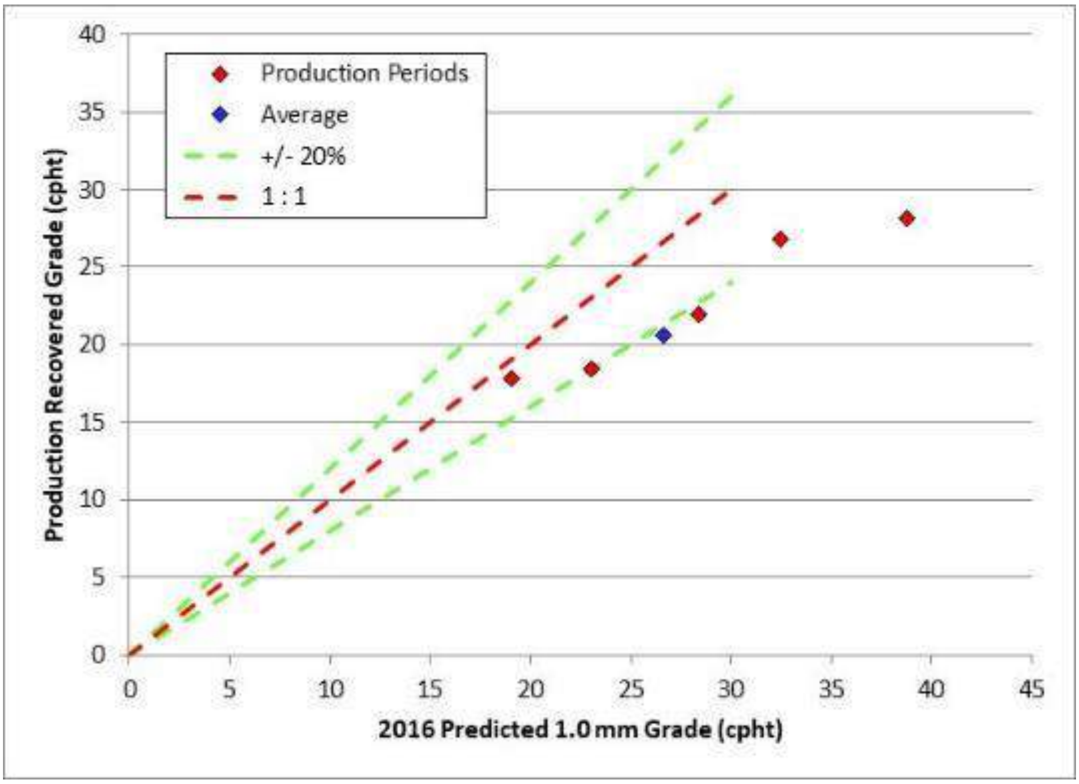


Figure 14.14 – Comparison of recovered vs block model grade for selected production periods

Comparison of recovered grade (cpht; 1.25 mm bottom cut-off) versus the block model grades (incorporating grades updated in 2016) predicted 1.0 mm grade (cpht) for five selected production periods. See text for details

Table 14.13 - Reconciliation of corrected block model with production and stockpile material

Data Source	Dry tonnes	Grade (cpht) (+1.25 mm)	Carats (+1.25 mm)	Comment
2013 Block Model	3 247 908	14	467 242	Predicted tonnes excavated and grade / carats recovered between 19 Nov 2012 and 21 Oct 2013. +1 mm grade estimates have been corrected by 25% to produce "recoverable" grade estimates at 1.25 mm.
Plant Production	2 065 176	21	426 954	Production data from Karowe plant from 19 Nov 2012 to 21 Oct 2013 (1.25 mm bottom cut-off). Wet tonnage corrected by 5% for moisture content.
Estimated Stockpile	1 070 236	4	40 288	Estimated tonnage mined (from block model) minus the plant production tonnage, corrected for internal basalt reef. Estimated carats and grade derived from 25% correction to +1.0 mm grade model minus production carats.

Reconciliation of corrected 1.0 mm block model grades (extracted from the block model following updates for revised grade estimates in 2016) with production data. The 25% correction applied to convert 1.0 mm block model grades to "recoverable" grades by the Karowe Mine produces a realistic estimated grade of 4 cpht for the material stockpiled during this period. By comparison, if no correction is made the calculated grade of material stockpiled during this period would be 16 cpht.

14.3.4 Confidence level of grade estimates

As indicated in Section 14.3.1 the LDD sampling provides a representative spatial distribution of mini-bulk samples across the AK6 kimberlite to depths of approximately 400 m below surface (600 mamsl). This provides a basis for local estimation of grade for this portion of the body at a level of confidence that is appropriate for classification as an indicated mineral resource. Grade is significantly less well constrained in the deeper portion of the body, between 604 mamsl and the base of the model at 260 mamsl. However, the demonstrated continuity of kimberlite units to depth provides a reasonable basis for the global estimates of average grade in this portion of the deposit, and for classification of this material as an inferred mineral resource. For practical purposes the bottom depth of the indicated resource has been rounded off at 600 mamsl. This adjustment is not considered to have any meaningful implication for overall grade uncertainty in the indicated portion of the deposit.

The grade models defined based on LDD samples represent diamonds recoverable by RC drilling and DMS processing with a bottom cut-off of 1.0 mm. As discussed in the previous section, conversion of the 1.0 mm grade model to a model of grade recoverable by the current Karowe plant (at a bottom cut-off of 1.25 mm) is not straightforward and introduces additional uncertainty to the final grade estimate. However, the maximum extent of uncertainty associated with the derived correction factor (25%) is considered to be of the order of ± 10%, considered by MSC to be an acceptable level of confidence for an indicated mineral resource.

14.4 Size Frequency Distribution (SFD)

Size Distribution Models were updated in 2013 (Lynn et al., 2014) and on an ongoing basis to reflect production parcels from each of the three lobes of the AK06 kimberlite. The production SFD's were defined by isolating production batches from each of the three lobes and developing appropriate SFD models thereof. The updated SFD models based on production were also used to derive correction factors for determining recoverable grade estimates based on the 1.0 mm grade model obtained by interpolation of LDD data (see Section 14.3 above, Lynn et al., 2014). The recoverable grade model (2013) derived from the production SFD data is considered to still be valid.

Production at the Karowe Mine is organized into batches based on exports of diamond parcels. Ore movement is maintained at the Karowe Mine through records of truck haulage which include material origin, tipping point and the type of material. Stockpiling of feed on the basis of lobe was instituted in 2014, additionally as DMS yield became a processing constraint, prior to implementation of Phase 2 and 3 plant upgrades, stockpiles were generated on basis of lobe, yield and grade. Individual stockpiles can be tracked back to source ore polygons. This has allowed for the assignment of material processed during specific production periods to the lobe from which it was sourced. Period of mixed feed are not assigned to specific lobes and therefore not utilised for SFD analysis.

Since 2012 over 2 million carats have been recovered from AK6 (Table 6.1) and the size distribution models for each lobe has been updated to reflect production data.

The 2017 SFD models are presented in Table 14.14 with South Lobe Sales data 2015-June 2017.

Table 14.14 – SFD Models

Size	North Lobe	Centre Lobe	South Lobe	South Sales
+3 DTC	0.13%	0.60%	2.46%	1.41%
+5 DTC	8.01%	11.33%	15.92%	15.10%
+7 DTC	9.24%	9.73%	11.64%	10.28%
+9 DTC	15.92%	15.56%	14.63%	15.40%
+11 DTC	24.68%	22.78%	17.58%	18.49%
3-6 Gr	25.73%	20.60%	16.80%	18.05%
8-10 Gr	7.70%	6.34%	5.58%	5.47%
3-5 ct	5.28%	5.84%	4.62%	5.94%
6-10 ct	2.37%	3.79%	4.08%	2.99%
+10.8 ct	0.95%	3.36%	6.62%	6.88%

For the north and centre lobes the 2017 SFD models display slight variance to 2013, the most significant variance is the weight proportion assigned to the +10.8ct size class for the South Lobe. The 2013 SFD model assigned 3.73 wt.% to the +10.8ct size category, the 2017 SFD models assigns 6.62 wt.% to this size category on the basis of production records. The incidence of specials increases with the proportion of South Lobe kimberlite feed to the process plant. Overall LOM production has 5.5 wt.% specials. (Figure 14.15)

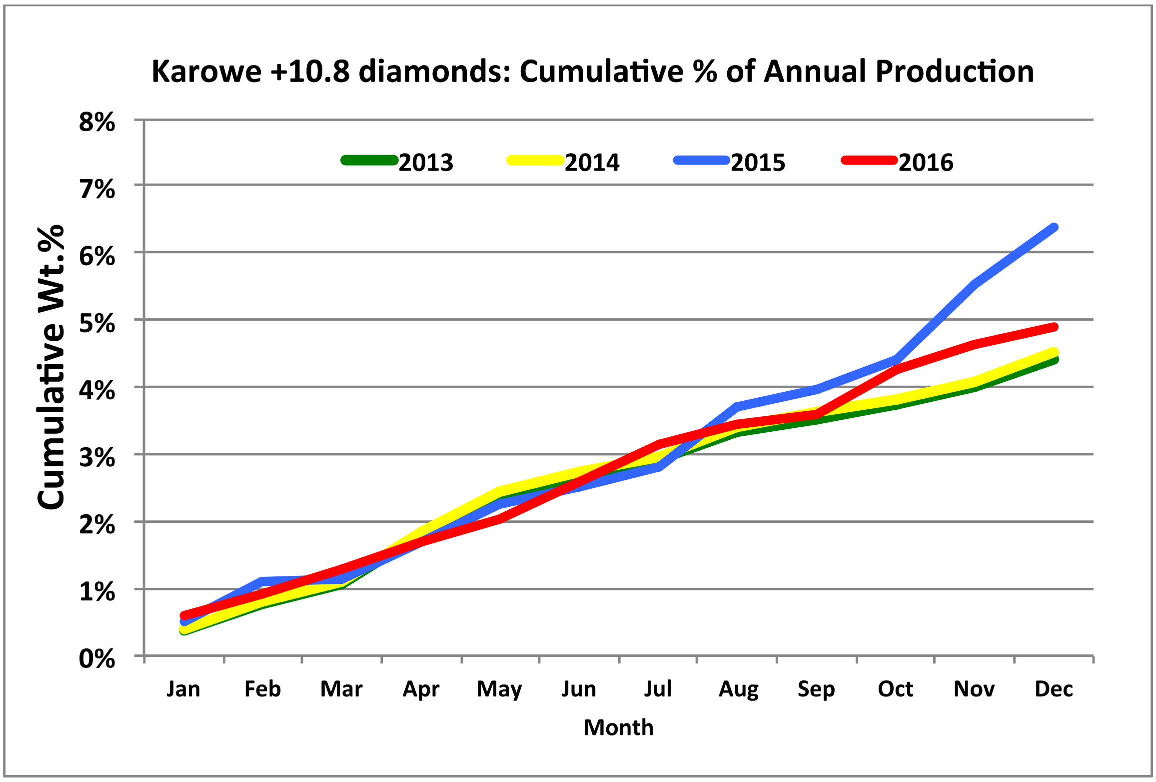


Figure 14.15 - Cumulative weight percent specials as a percentage of annual carat production

Weight percent +10.8ct for each production quarter, indicating the variability and influence of south lobe production (Figure 14.16).

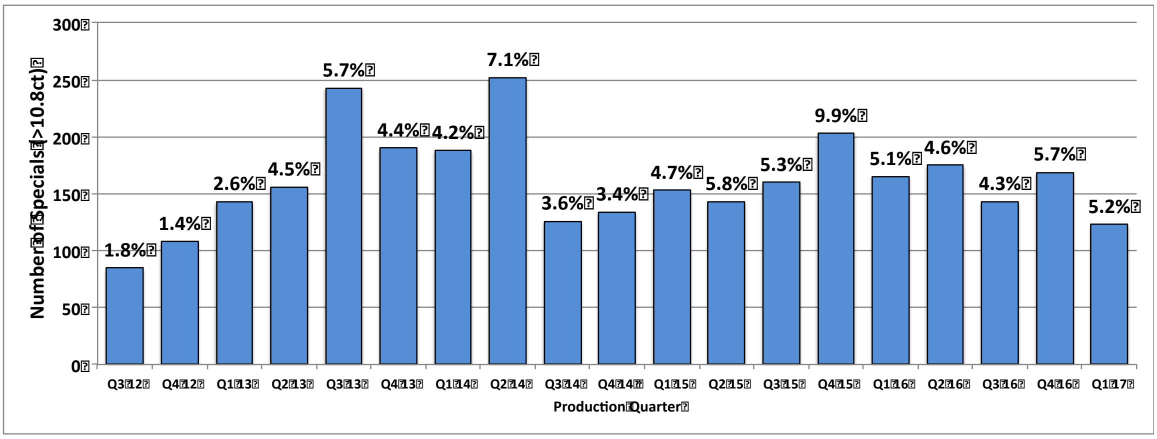


Figure 14.16 - Weight percent specials (+10.8 carat) and stone count for quarterly carat production

14.4.1 SFD continuity

The LDD data for AK6 were investigated by lobe, by elevation within each lobe and by rock type within each lobe to evaluate variations in SFD. Due to the relatively small size of the diamond parcel recovered from LDD samples (1,175 ct), it is not possible to reliably evaluate variations in SFD on the scale of sampling. However, by grouping the LDD samples into larger datasets meaningful observations can be made regarding variation in SFD with depth and between lobes.

The SFD’s of LDD diamond parcels grouped by lobe are shown in comparison with selected production SFD data isolated by Lobe from early production in 2013 (see Lynn et al., 2014) in Figure 14.17 below. Due to the small size of the LDD parcels, the coarse end (> ~1.5 ct) of the size distribution is not reliably represented in the LDD datasets, in particular those for the Centre and North Lobes. In addition, due to different liberation and recovery parameters, the SFD curves representing LDD parcels are not directly comparable to those derived from production data. Nonetheless, aspects of the SFD data for the LDD parcels correlate well with those defined by production results for the same lobes. The overall size distribution differences observed in the production data for each lobe are broadly reflected in the LDD parcels and, despite the fact that the coarse end of the distribution is not well represented in the LDD dataset, the relative proportions of very coarse diamonds observed in the production batches from different lobes are reflected in the LDD data. This indicates that variations in the recoverable SFD between different portions of the AK6 deposit are reflected in the LDD diamond size data, suggesting that the LDD data can be used to assess the potential for variation in SFD with depth, providing sufficient data are available to support this.

To evaluate variation in SFD with depth, the LDD data for the South and Centre Lobes were grouped based on the elevation range from which each sample was derived (Figure 14.18). While it is not possible to evaluate SFD variations on a local scale (e.g. by block or mining level), the results suggest that there is no significant overall change in SFD characteristics with depth on a large scale (e.g. between the upper and lower portions of each lobe). Similarly, grouping of LDD data by kimberlite unit provides no indication of significant variations in SFD between the main unweathered kimberlite units of the South or Centre Lobes. Due to the different recovery characteristics of weathered kimberlite as well as the small size of LDD parcels derived therefrom, it is not possible to reliably interpreted variations between equivalent weathered and fresh kimberlite units.

In conjunction with the confirmation of geological continuity discussed in Section 14.1.2, the analysis described above supports the use of SFD data for kimberlite production from shallow levels in each lobe to represent all kimberlite within that lobe.

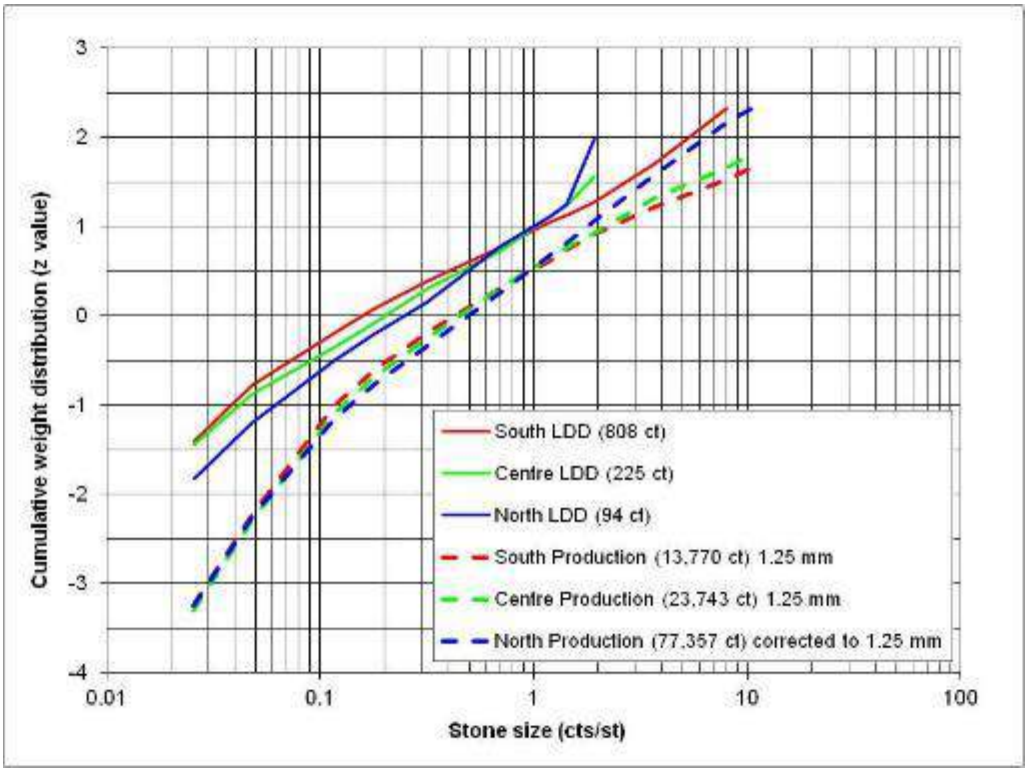


Figure 14.17 – Cumulative log probability plot of LDD SFD in comparison with production SFD

Cumulative log-probability plot illustrating the SFD of LDD results by lobe in comparison with production data by lobe; plot shows the proportion of diamonds by weight below a given stone size. The Centre Lobe SFD has been corrected by excluding a single +13 ct diamond as a statistical outlier

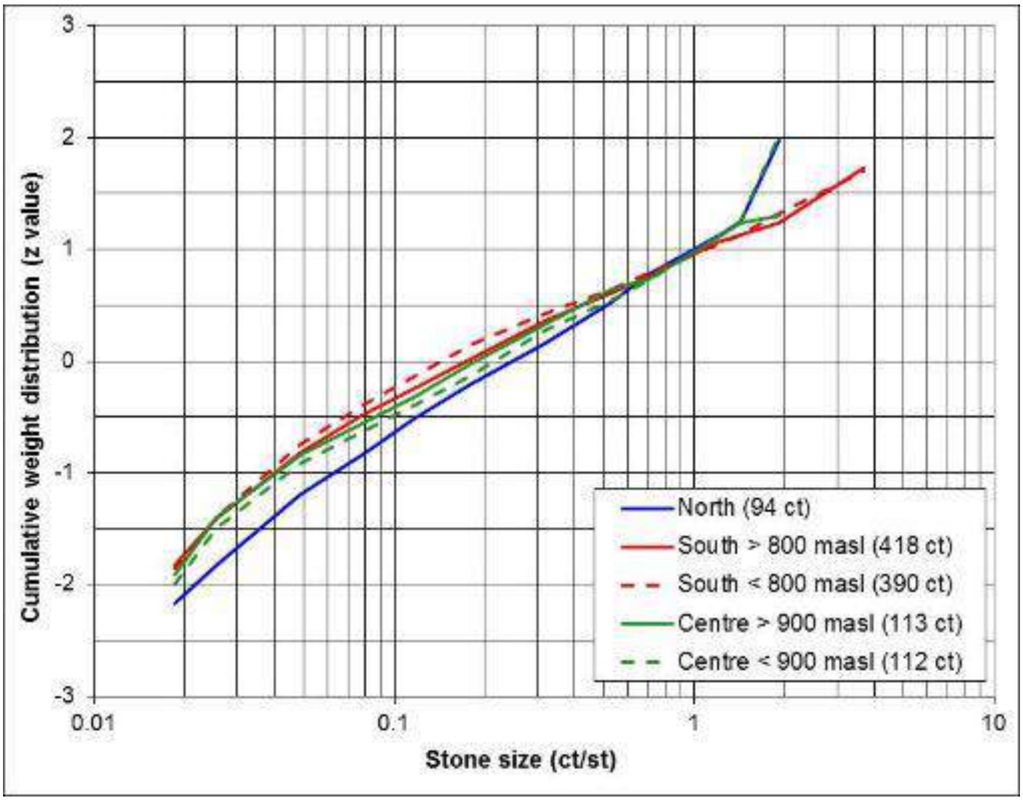


Figure 14.18 – Cumulative log probability plot of LDD SFD grouped by depth range

Cumulative log-probability plot illustrating the SFD of LDD recoveries grouped by depth within the South and Centre Lobes; plot shows the proportion of diamonds by weight below a given stone size. The SFD of the North Lobe LDD parcel is shown for reference.

14.5 Diamond Value

In excess of 1.8 million carats of diamond produced from AK6 have been sold up to June 2017 for revenues of US$1.11 billion for an average price of US596$/ct (Table 6.1). A population of high value diamonds is present in the AK6 diamond population. Based on production records high value coloured and white (colourless) diamonds have been recovered from each of the three lobes. Certain large and /or high value diamonds have been sold in Exceptional Stone tenders (EST's), the reserve price for these goods has increased as the recovery of these diamonds has increased in proportion to total production. Individual high value diamonds are also sold during regular tenders.

To analyse and establish price models, production data per source lobe is utilised using similar criteria as for SFD analyses. Production records and records for recovery dates for specials (+10.8ct diamonds) are used to determine the source lobe for production parcels prior to rolling of diamonds into sales lots. The Lucara price book is applied to the sorted goods prior to rolling into sales lots, all single diamond lots ("specials") are assigned reserve prices based on estimated polished outcomes and treated individually for pricing. Once diamonds sale tenders are completed the actual achieved prices for single stone sales are reconciled back into the 'reserve price' for each individual production parcel. Price models for each lobe do not include high value diamonds. The impact of high value stones is significant within the Centre and South lobes due to the coarse nature of the realized and modelled SFD's. Removal of the high value diamonds from the value models mutes the impact of this high value aspect of the AK6 production.

Price models for each lobe are presented in Table 14.15, models are based on Lucara's 2017 rough diamond price book data and achieved prices for diamonds sole as individual lots. Value models are based on a BCOS of 1.25mm.

Table 14.15 – Price Models by Lobe

Size	North Lobe	Centre Lobe	South Lobe
+3 DTC	$ 35	$ 42	$ 40
+5 DTC	$ 47	$ 47	$ 47
+7 DTC	$ 56	$ 56	$ 56
+9 DTC	$ 64	$ 65	$ 65
+11 DTC	$ 95	$ 95	$ 95
3-6 Gr	$ 222	$ 209	$ 188
8-10 Gr	$ 484	$ 420	$ 430
3-5 ct	$ 808	$ 667	$ 832
6-10 ct	$ 1 127	$ 1 187	$ 1 190
+10.8 ct	$ 1 600	$ 6 063	$ 8 250
AP US$ /ct	$ 222	$ 400	$ 730

A real rough diamond price escalation of 2.5% per annum has been applied to the average price models for the south lobe for the purposes of the PEA.

14.6 Mineral Resource Statement

The estimates of kimberlite volume, bulk density, tonnage, grade and average diamond value described in the sections above have been integrated to generate a mineral resource estimate for the AK6 kimberlite, presented in Table 14.16. Estimated tonnes and carats reflect the depleted resource, with material mined up to the end of December 2016 removed from the original model. Resource grade and average value estimates (updated in 2016 and 2017, respectively, from those reported in Lynn et al., 2014) reflect expected recoverable diamond production using the 2013 Karowe plant configuration with a bottom cut-off of 1.25 mm. Although significant changes to the plant configuration have been undertaken the recoverable grade model correction factor used in 2013/16 is considered still valid. The AK6 mineral resource estimate is reported by lobe and by mineral resource classification. Classification is based on CIM guidelines for reporting of mineral resources (CIM, 2010).

Resources are reported as those remaining as at end December 2016 and do not account for subsequent mining depletion. For reasons outlined in the sections above, the upper ~ 400 m of the deposit (to an elevation of 600 mamsl) has been classified as an indicated mineral resource, comprising an estimated total of 37.7 million tonnes of kimberlite ore, containing 5.53 million carats of diamonds at an average diamond value of $666 per carat

The portion of the deposit from 600 mamsl to the base of the model at 260 mamsl is classified as an inferred mineral resource, with an estimated total of 20.57 million tonnes of kimberlite ore, containing 2.98 million carats of diamonds at an average diamond value of $727 per carat.

Stockpiles at the Karowe Mine as of the 31st December 2016 were estimated to contain 4.53 million tonnes of kimberlite classified as an indicated mineral resource and containing 0.33 million carats of diamonds at an average diamond value of $465 per carat. These estimates are based on stockpile inventories maintained by Karowe Mine

Table 14.16 - Resource statement for the AK6 kimberlite

Resource	Volume (Mm^3)	Density (tpm^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	$/ct
North Lobe	0.64	2.48	1.58	0.24	15	221
Centre Lobe	1.68	2.57	4.30	0.70	16	400
South Lobe	11.10	2.87	31.82	4.59	14	730
Working SP	1.23	1.87	2.30	0.25	11	437
LOM SP	1.19	1.87	2.23	0.08	4	547
Indicated Total	**15.84**	**2.67**	**42.23**	**5.86**	**14**	**655**
Centre Lobe	0.08	2.58	0.20	0.03	15	400
South Lobe	6.87	2.96	20.37	2.95	14	730
Inferred Total	**6.95**	**2.96**	**20.57**	**2.98**	**14**	**727**

Statement of the estimated remaining mineral resource in the AK6 kimberlite deposit as of the end December 2016. Volume, tonnes and carats are reported in millions (M), and reflect updates made to the grade and value models in 2016 and 2017, respectively.

15 Mineral Reserve Estimates

Due to the preliminary nature of this report, this Preliminary Economic Assessment report does not state a Mineral Reserve.

16 Mining Methods

16.1 Geotechnical Assessment

16.1.1 Data sources and previous study work

Several historic geological and geotechnical reports were made available from which to extract relevant data and gain initial understanding. These are dated from August 2006 to January 2017 and focus predominantly on the open pit design areas. A full list of these documents is contained in Section 27 References.

- Barnett (2007) provides details on the geological 3D model still in effect at the time of this PEA. A revision of the model is currently underway and should be available for use during the next stage of study.
- Armstrong and Venter (2007) and Ekkerd and Ruest (2008) both provide results of laboratory rock strength testing and Rock Mass Ratings which has informed the basis of the decision making for the PEA (as summarised in Table 16.4). Laboratory testing of samples from the latest drilling that target the underground mine areas should be available for use during the next stage of study. SRK consulting (Johannesburg) is undertaking a review of the geotechnical database / model.
- Bush et.al. (2017) completed a Geotechnical and Hydrogeological review of the Karowe open pit and includes results from the three geotechnical "GT" holes drilled during 2016 / 2017.

Fifteen new delineation holes were drilled in 2016 / 2017. Drilling and collar details are shown in Table 16.1. Seven of the 15 holes had triple tube core recovery suitable for geotechnical investigations. These are marked with a light green cell background in Table 16.1 and have formed the basis of assessment during the PEA. Three of the triple tube holes, annotated with "GT" in the BHid, were specifically drilled for geotechnical reasons to investigate the weak red mudstone bands in the Mosolotsane Sandstone unit as well as to collect information on the underground mining target area. The following data was available for these drill holes:

- Drill core logs are available for each of the 15 holes in Excel format and include Geotechnical measurements.
- Core photographs of all 15 holes are available albeit lacking the usual geotechnical markings to allow for photographic checking of natural joints/breaks.
- Geophysical (Televiewer) survey reports are available on 6 of the holes GT2a, GT3 REP004, REP006B, REP009 and REP012.
- Laboratory testing of rock strength is currently underway.

The Barnett (2007) version of the 3D geological model (with lithology's and major contacts) appears to be reasonable at this level of study but lacks sufficient detail to allow for geotechnical zonation of the main geological members necessary for detailed geotechnical design purposes.

Point load index (PLI) and rock density test work is available for the Kimberlite lobes and is limited to the current open pit depth (60 to 120m below surface).

In Conclusion, the geotechnical database available during the PEA study comprised the following:

- 3D geological contact geometry as referred to by Barnett (2007). Bush et. al. (2017) makes reference to an August 2013 update of this model. A revision of the geological model is however currently underway and should be available for use during the next stage of study.
- Structural model after Barnett (2007) focussed on the definition of fracture zones.
- Joint information (spacing and quality) and the rock mass parameters after the 2016 / 2017 drilling programme. A detailed assessment of this information is currently underway and aims at the creation of a complete geotechnical database / model.
- Rock strength parameters from past geotechnical reports as described above. Laboratory testing of samples from the 2016 / 2017 drilling should be available for use during the next stage of study.

Table 16.1 - Details of 15 exploration drill holes that form the basis of current understanding

DHid	Date First Drilled	Date first Logged	Collar (m)			Max Depth (m)	Hole Orientation (Avg)			Detailed assessment during
			Easting	Northing	Elevation		Trend (000°)	Plunge (00°)	N-E-S-W	
GT1a	3-Feb-16	17-Mar-16	341,319.2	7,621,475.8	1,013.4	742.5	044	-55	NE	PEA and PFS
GT2a	19-May-16	7-Jul-16	341,777.3	7,622,090.2	1,012.3	901.6	207	-55	SSW	PEA and PFS
GT3	9-Mar-16	10-Jun-16	341,916.2	7,621,503.0	1,013.3	874.9	298	-61	NW	PEA and PFS
REP001	13-Jun-16	27-Jul-16	341,110.3	7,621,702.1	1,013.7	854.4	095	-49	East	PFS
REP002	16-Jun-16	16-Aug-16	341,579.3	7,622,200.2	1,011.7	800.6	188	-43	South	PFS
REP003	16-Jun-16	31-Aug-16	341,553.3	7,621,337.5	1,014.0	806.9	353	-55.5	North	PEA and PFS
REP004	30-Jul-16	12-Oct-16	341,743.0	7,622,089.3	1,015.0	893.4	??	-50	SSW	PFS
REP005	30-Jul-16	21-Sep-16	341,628.7	7,622,167.9	1,011.8	758.4	201	-40	SSW	PFS
REP006B	16-Nov-16	1-Feb-17	341,270.0	7,622,220.5	1,012.1	917.4	156	-44	SE	PEA and PFS
REP007	4-Sep-16	17-Oct-16	341,939.2	7,621,890.8	1,012.0	818.3	246	-54	WSW	PFS
REP008	15-Sep-16	2-Nov-16	341,229.7	7,621,750.7	1,013.5	755.4	088	-57	East	PFS
REP009	22-Sep-16	22-Nov-16	341,073.8	7,621,739.6	1,013.6	917.9	101	-55	East	PEA and PFS
REP010	20-Oct-16	25-Jan-17	341,939.0	7,621,892.0	1,012.0	809.4	??	??	WSW	PFS
REP011	21-Oct-16	8-Dec-16	341,228.7	7,621,749.2	1,013.6	668.4	112	-48	ESE	PFS
REP012	3-Dec-16	16-Feb-17	341,941.3	7,621,880.4	1,011.8	753.0	249	-49	WSW	PEA and PFS

16.1.2 3D Geological model and geotechnical setting

The 3D Geological model after Barnett (2007) is shown in Figure 16.1 with additional detail in Table 16.2. Nine (9) country rock units were differentiated along with three (3) kimberlite lobes. The south kimberlite lobe is the largest of the three and is the target of planned underground mining between 320 mbs to 590 mbs.

Each of the main geological units are described in terms of its geotechnical relevance below.

Kalahari Sediments (Sand and Calcrete)

These consist of sand, silcrete and calcrete, varying in thickness between 10m and 40m. The 2007 model suggests a thin unit of sand / soils which should not impact severely on planned mining operations. The Calcrete is reasonably competent for an upper geological unit having a UCS of 111 MPa and extends to a thickness of 10-15 metres. Risk mining through this unit is considered low but localised drilling will be required at the feasibility stage to guide the placement of surface infrastructure related to the underground operations.

Stormberg Basalt

The basalt is amygdaloidal, made up of layers of jointed basalt flows, with a total thickness of about 120m based on the 2007 model. Weathering has been observed between flows and along jointing. The unit has been sub divided into a shallower predominantly weathered zone (10 to 15 metres thick) underlain by more competent fresh material. Experience during open pit mining at Karowe shows the unit to be competent when good blasting practice is followed. This unit is unlikely to influence stability of the underground mine but its role in the interaction between the opencast and underground operations warrants further investigation.

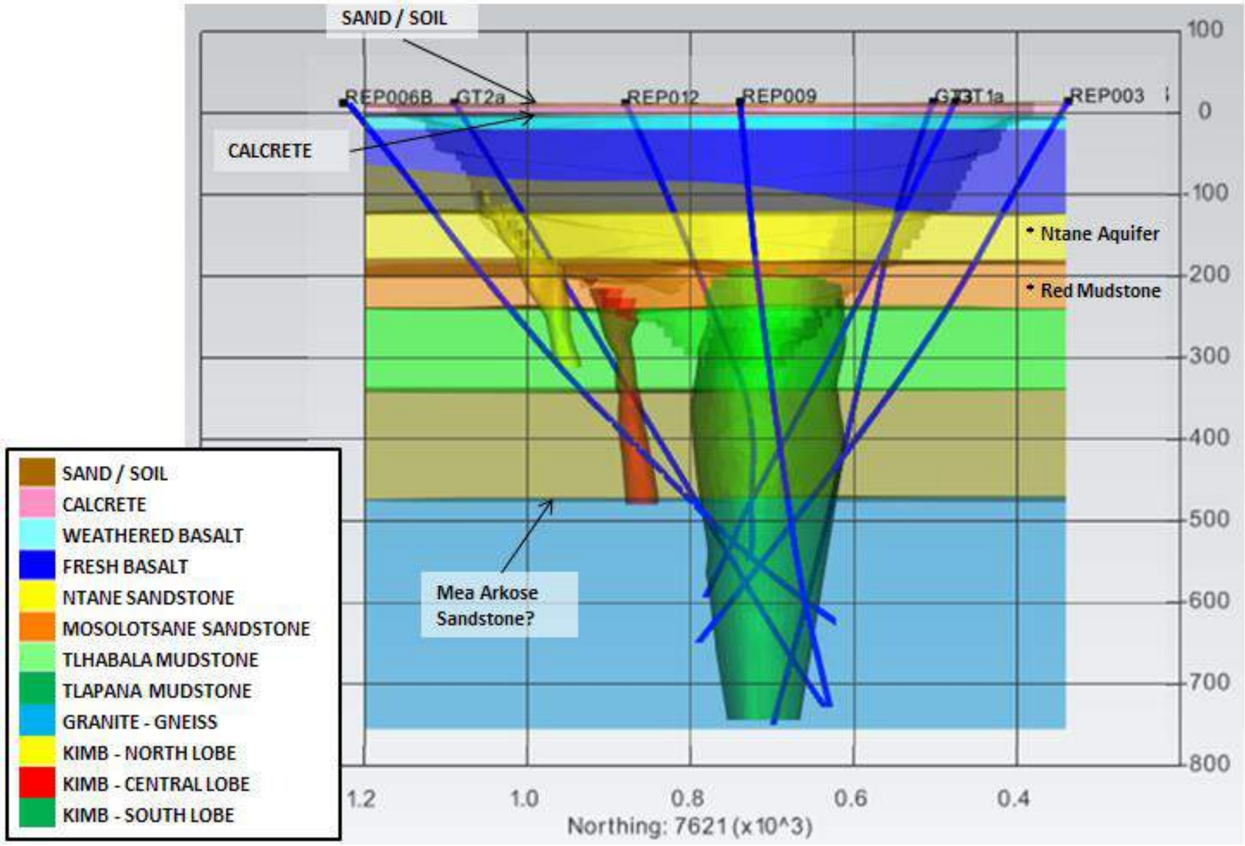


Figure 16.1 - 3D Geological model after Barnett (2007)

Table 16.2 - Basic 3D model geological units taken verbatim from Barnett (2007)

Formation	Dominant Rock Type	Modelled Thickness (m)	Top contact guideline
Stormberg	Basalt	117 - 127	Basalt present
Ntane	Sandstone	55 - 100	Sandstone replaces basalt
Mosolotsane	Sandstone with minor mudstone	33 - 61	1st occurrence of mudstone
Lekotsane	Sandstone	0	
Tlhabala	Mudstone with minor sandstone	92 - 107	Change to dominantly mudstone
Tlapana	Carbonaceous mudstone	127 - 139	1st occurrence of graphite bearing sediments.
Mea	Sandstone	0	
Basement	Granite Gneiss		Change to gneiss

Ntane Sandstone

The Ntane formation is a major aquifer in Botswana and is the main source of water supply for the Orapa / Letlhakane mine areas. It is a fine to medium-grained sandstone primarily of aeolian origin. The thickness of the Ntane at Karowe varies from 55m to 100m tending to be thickest at the north-eastern extremity of the mine.

The unit is weak having an average UCS of 33 MPa mostly due to its low density (2.0 t/m^3) and high porosity. Stability when developing through this unit will rely on proper dewatering and on the avoidance of highly stressed areas (i.e. staying clear of the final cave and relaxation zones).

Mosolotsane Formation

The Mosolotsane is a sequence of fine to coarse-grained sandstones, siltstones, and reddish-pink mudstones that were deposited in an alluvial environment. According to the Barnett (2007) model the unit varies in thickness from 33m to 61m. The formation consists of two relatively distinct units:

- an upper mudstone unit consisting of inter-bedded mudstones, siltstones, and fine to medium-grained sandstones, and
- a basal, coarse-grained, locally conglomeratic, arkosic sandstone.

The unit is stronger than the Ntane having a density of 2.3 t/m^3 and an average UCS of 50 MPa. Inter-collated bands of red mudstone occur within the Ntane, which degrade readily when exposed to water. Accounts from Bush et.al. (2017) indicated that the red mudstones are particularly susceptible to weathering, with entire core sections having weathered to soil and gravel on exposure. These exposures in the core are sealed in cling wrap in an attempt to retain sample integrity. An example from drill hole GT3 is shown in Figure 16.2.

At Karowe studies of the GT drill holes have shown that the red mudstone bands have an average vertical thickness of 5m to 8m with the maximum vertical thickness of 17m measured in drill hole GT1a. Development through these localised mudstone areas will be difficult and slow requiring strategies for extensive permanent support.

Figure 16.2 - Decomposed red mudstone from drill hole GT3.

Tlhabala Massive Mudstone

The Tlhabala Formation consists of massive grey shales and mudstones with a thickness at Karowe of 92 m to 107 m. The unit provides a reasonable strength with a density of 2.3 t/m^3 and an average UCS of 87 MPa. The range in UCS observed at Letlhkane Mine (23 km to the east) range from 35-40 MPa and reportedly are prone to weathering which results in difficult mining conditions. The mudstones at Karowe do display differences to those reported at Letlhakane Mine.

The Karowe mudstone is less susceptible to weathering and remains relatively intact in the core, although some localised examples of similar degradation to the red mudstone are observed (Bush et.al. 2017). Ekkerd and Ruest (2008) report that results of weathering analysis (Figure 16.3) indicated that the mudstone unit at Karowe is not prone to short term weathering. This lower weathering rate could be attributed to low smectite content in this area of the sedimentary basin. Current durability testing samples from the 2016 / 2017 drilling programme is underway and will be available to augment the 2008 data during the next stage of study.

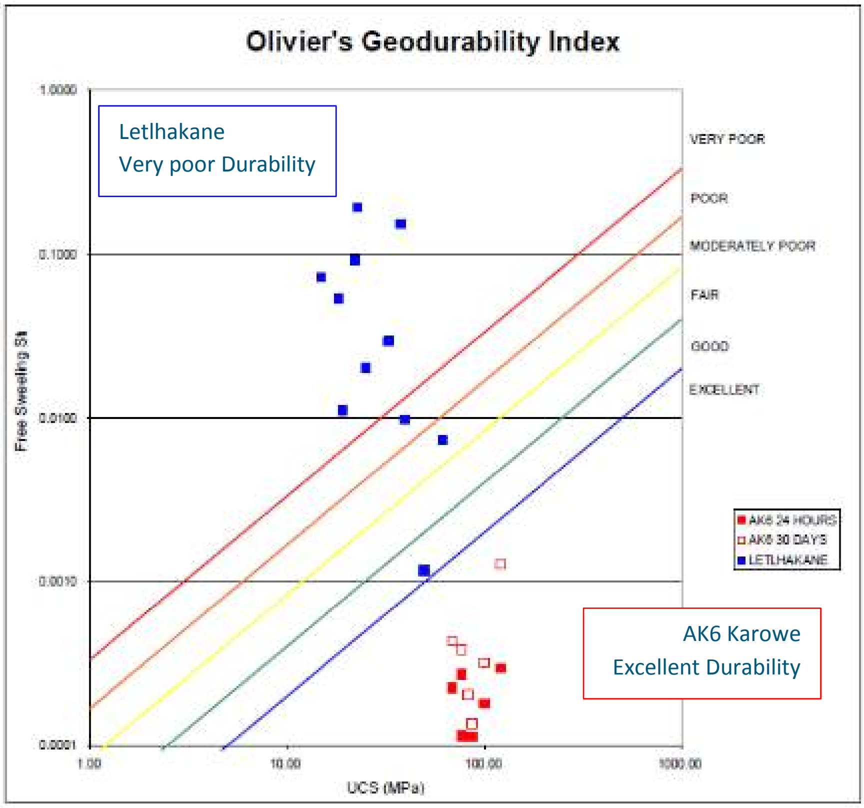


Figure 16.3 - Laboratory determined durability indices of mudstones contrasting the Karowe and the Letlhakane mine regions (after Ekkerd and Ruest, 2008)

Further investigation of the Tlhabala mudstone is required to better understand the weathering and competency of the unit. According to Bush et.al. (2017), caution should be exercised in determining the physical properties as they are likely to be controlled by the observed bedding in the core (anisotropic rock strength with a preferential direction of weakness).

Bush et.al. (2017) notes that the Tlhabala mudstone contacts appear to be an area of concern with very little solid core recovery across the contact zones. The weak contact and foliation should be investigated and considered in the geotechnical design.

Figure 16.4 - Decomposed Tlhabala mudstone contact after Bush et.al. (2017)

Tlapana Carbonaceous Mudstone

The Tlapana mudstone is a sequence of primarily carbonaceous mudstones with some thin coal seams and sandstone lenses. At Karowe, the average thickness of the unit is 127 m to 139 m. This is significantly thicker than is typical of neighbouring mines. Less is known about this unit at Karowe as it is yet to be exposed by the open pit and published investigation work is therefore limited.

As with the Tlhabala mudstone, the Tlapana unit provides a reasonable strength with a density of 2.3 t/m^3 and an average UCS of 85 MPa (compared to 35-40 MPa at Letlhakane).

Drill hole GT3 is a particularly good example of extensive presence of carbon-rich material and coal and sandstone filled beds or lenses (Figure 16.5 shows an extensive coal-filled bed). This dark coloured weak fractured zone extends from 465 m to 535 m along the drill hole length. These areas are reported as being weak and moderate to highly fractured with smooth polished joint surfaces.

During a recent site investigation of core, it was noted that the weaker carbonaceous zones are vertically extensive in most of the drill holes – amounting to as much as 80 m in vertical thickness. The carbonaceous material is competent when recovered but quickly deteriorates as a very dense network of micro fracturing (now depressurised) relaxes. Further work is required to better understand these units including immediate out-of-the-hole rock strength testing and geotechnical site logging.

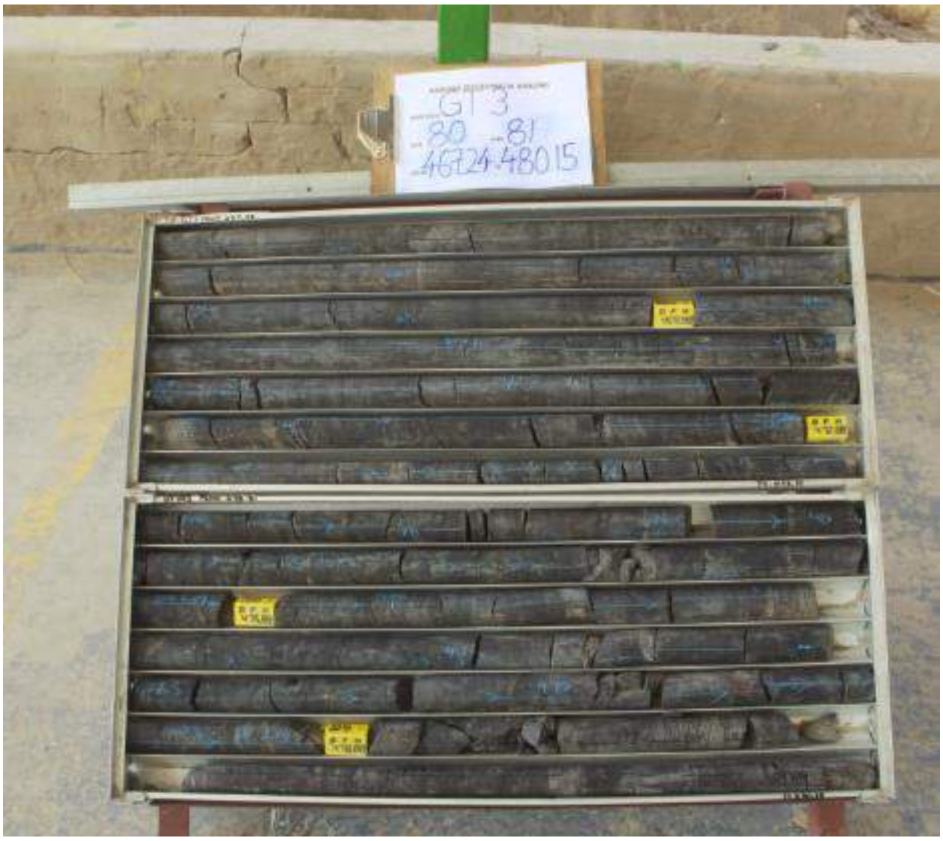


Figure 16.5 - Carbon rich mudstone with coal filled beds – weak fractured zone with polished joint surfaces

For the PEA it is assumed that development will remain challenging and that extensive ground support systems will be required.

Mea Arkose

This is a fine-grained to granular and pebbly arkosic sandstone whose thickness is not well defined, but is believed to be highly variable and ranges from about 25 m to 60 m at other mines (KLMCS, 2010). The contact between the Mea Arkose and the overlying Tlapana Mudstone appears in places to be gradational and not sharply defined.

The extent of and the hydrogeological properties of the Mea Arkose at Karowe requires confirmation.

Basement Granite / Gneiss

The basement rocks are collectively referred to as granite, but whose composition also includes, in addition to true granites, granite gneisses, amphibolites and ultramafic schists. At Letlhakane mine the upper 30 m of the granite unit (the so-called granite contact zone) has yielded artesian water continuously since 1980 with estimated hydraulic conductivity as high as 0.15 m/day to 0.55 m/day (Morton, 2010). Core drilling at Karowe has intersected the granite and upper granite in 6 drillholes, artesian water has not been reported or observed.

Typical values of UCS for the granite are in excess of 200 MPa and from a geotechnical stand point the unit appears (visually) competent. However, density and UCS values must still be measured through laboratory test work. In proximity to the kimberlite contact, the granite host may weaken with core visually more fractured. An example is shown in Figure 16.6. This area falls within the kimberlite contact zone described below.

Figure 16.6 - Weakened Brecciated Granite found at the Granite – Kimberlite contact (GT3)

Kimberlite

The Karowe kimberlite pipe is an opaque mineral-rich monticellite kimberlite texturally classified primarily as fragmental volcanoclastic kimberlite with lesser macrocrystic hypabyssal facies kimberlite of the Group 1 variety. The South Lobe is considered to be distinctly different from the North and Centre Lobes, which are similar to each other in terms of their geological characteristics.

According to Ruest (2008) the South Lobe overall contains low levels of crustal dilution and below the weathered horizon is characterised by an abundance of fresh unaltered olivine macrocrysts and phenocrysts, as well as groundmass monticellite (Ca-olivine). The South Lobe is an extremely competent kimberlite with a density of 2.9 t/m^3 and a high UCS averaging 156 MPa.

This high density / strength reflected in the geotechnical lab results is consistent with the past and present geological observations. However, additional test work will be undertaken. Implications for the mine design are significant in that the unusually high density and strength may affect overall caveability.

The pipe at depth consists of two major facies, the MPK-S which is unaltered and the less abundant EMPK-S kimberlite, which is diluted, on the eastern extremity of the south lobe (Figure 16.7).

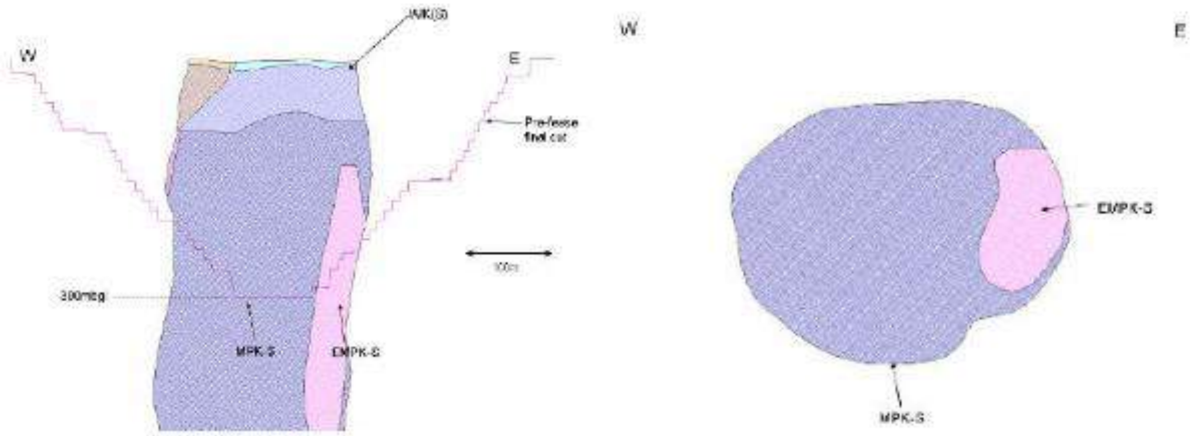


Figure 16.7 - Geometry of the South kimberlite lobe differentiating between unaltered MPK-S and diluted EMPK-S (after Ruest, 2008)

Kimberlite contact zone

The kimberlite contact zone is typically associated with deterioration of ground conditions and increased stability risk.

Geologically the contact zone between the country rock and the kimberlite pipe is unbonded and can become a conduit for ground water flow. The contact can open, close, or shear depending on associated stresses created by nearby mining. Typically, contact dislocations open as stress is relieved by mining and the contact zone should then be de-watered to avoid the transmission of water into the kimberlite and to reduce resulting ground condition deterioration due to relaxation.

Within the country rock this is evident in the drill hole core as rubble zones or zones of notably increased fracturing extending 5 m to 7 m horizontally before the contact is reached. Local variability in the contact conditions between drill holes is illustrated by comparing GT1a and GT2a in Figure 16.8 below. Kimberlite dilution and brecciation (the EMPK-S kimberlite unit) also plays a role in contact zone ground conditions. This is more prevalent on the south-eastern side of the lobe.

Ground deterioration may be expected within the kimberlite although this is not readily evident from the core photographs. Kimberlite within the immediate contact area (typically 10-15m horizontally) is more intensely intersected by fine hydrothermally formed fractures normally with slickensided surfaces. This can cause increasingly blocky ground conditions.

Figure 16.8 - Comparing the kimberlite contact zones of GT1a and GT2a

Zone of Relaxation

As mining and caving progresses fractures develop in the cave walls and relax and open up towards the centre of the cave. A zone of relaxation (ZOR) develops, which in combination with the kimberlite contact zone, allows a zone of preferred ground water movement as well as tunnel deformations. Typically, the relaxation zone extends to about 100m horizontally from the pit perimeter (close to surface) and tapers vertically downwards to the width of the contact zone at the lowest underground extraction level. Best practice is that all storm water is diverted away from the ZOR and that critical LOM infrastructure is located outside of it.

Empirical methods have been devised to assess the expected break back and zone of relaxation around a cave. These models consider mining depth, height of caved material, rock mass conditions (RMR) as well as the lobe geometry. These methods will be interrogated in detail during the PFS.

Structural model

The structural model in use was constructed by Barnett (2007) from a borehole core investigation conducted in 2007 for project purposes, and focussed on the definition of fracture zones (Bush et.al. 2017).

This model, shown in Figure 16.9, has not been validated with in-pit observations. The current model is in need of revision as it is focused on fracture zones only and on the shallower open pit areas. A revision of the model is currently underway.

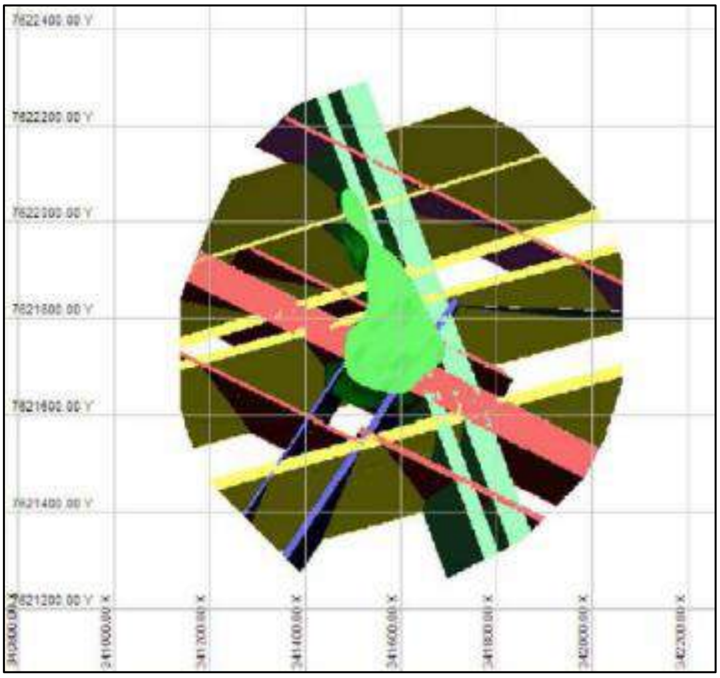


Figure 16.9 - Fracture zones associated with the AK6 Kimberlite Pipe (Barnett, 2007)

16.1.3 Geotechnical data from the 2016 / 2017 drilling programme

Collar locations and drill hole trends for the 2016 / 2017 drilling programme are shown in Figure 16.10. Figure 16.10 (a) shows all 15 drill holes while Figure 16.10 (b) shows only the seven triple tube drilled holes which formed the basis of visual assessments during the PEA. Detailed geotechnical assessments of the logs of all 15 drill holes are underway and will be used to inform the next stage of study (PFS).

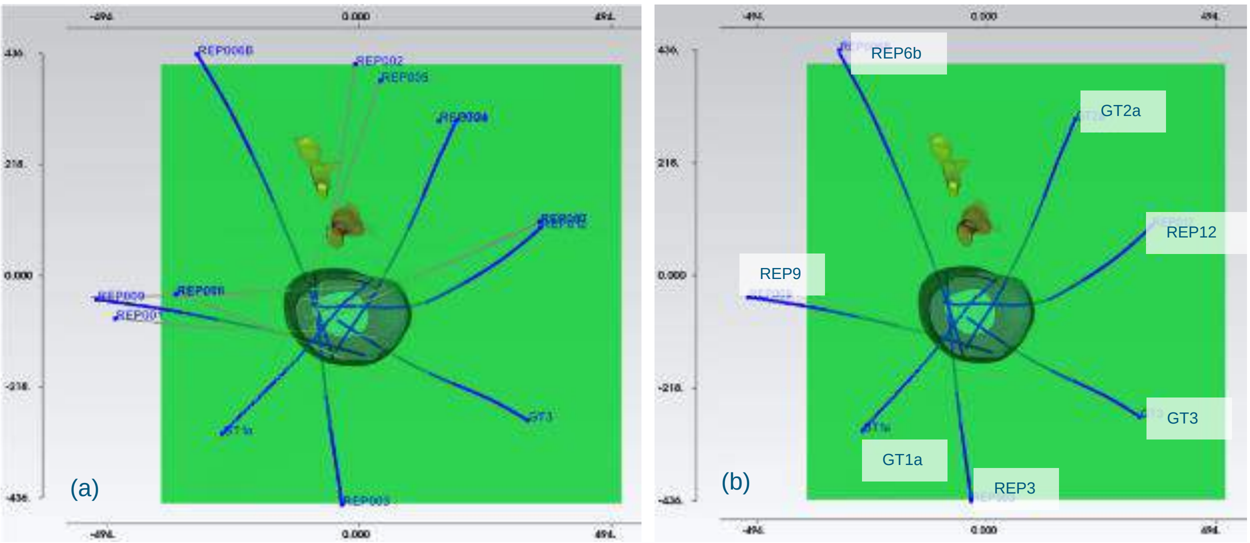
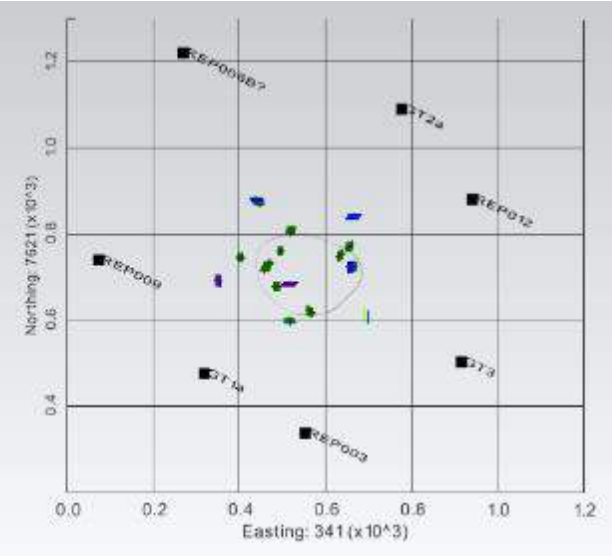


Figure 16.10 - Plan view focused on the south lobe showing (a) the locations of all the 2016/2017 boreholes and (b) the locations of the 7 triple tube drilled boreholes

The seven triple tube holes are well located around the perimeter of the targeted kimberlite and cover both the kimberlite and the country rock reasonably well between 650 and 450 mamsl (mining depth range 350 m to 550 m below surface).

The assessment of drill hole coverage is summarised by Figure 16.11. Additional borehole coverage is required in the country rock for mining depths below 550m below surface.



Elevation (mamsl)	Depth (mbs)	Host Rock	Ore body
700	313	Excellent	None
650	363	Excellent	Contact only
600	413	Good	Reasonable
550	463	Reasonable	Good
500	513	Poor	Good
450	563	Poor	Good
400	613	Contact only	Reasonable
350	663	Poor	Contact only
300	713	Poor	Poor
250	763	None	Poor
200	813	None	None

Figure 16.11 - Example of drill hole distributions at 600 mamsl or 413 mbs (LHS) and summary of coverage per elevation (RHS)

Figure 16.12 summarises the geotechnical model for Karowe as at completion of the PEA. Geological units are shown along with average values for density, UCS and RMR. The model is based on the Barnett (2007) model, which was later updated in 2013, and the geotechnical parameters are based on available reports as described in more detail below (Section 16.1.4). Geological units are the sole basis for the geotechnical units at this stage. Additional data may require modifications to the existing geotechnical unit definitions.

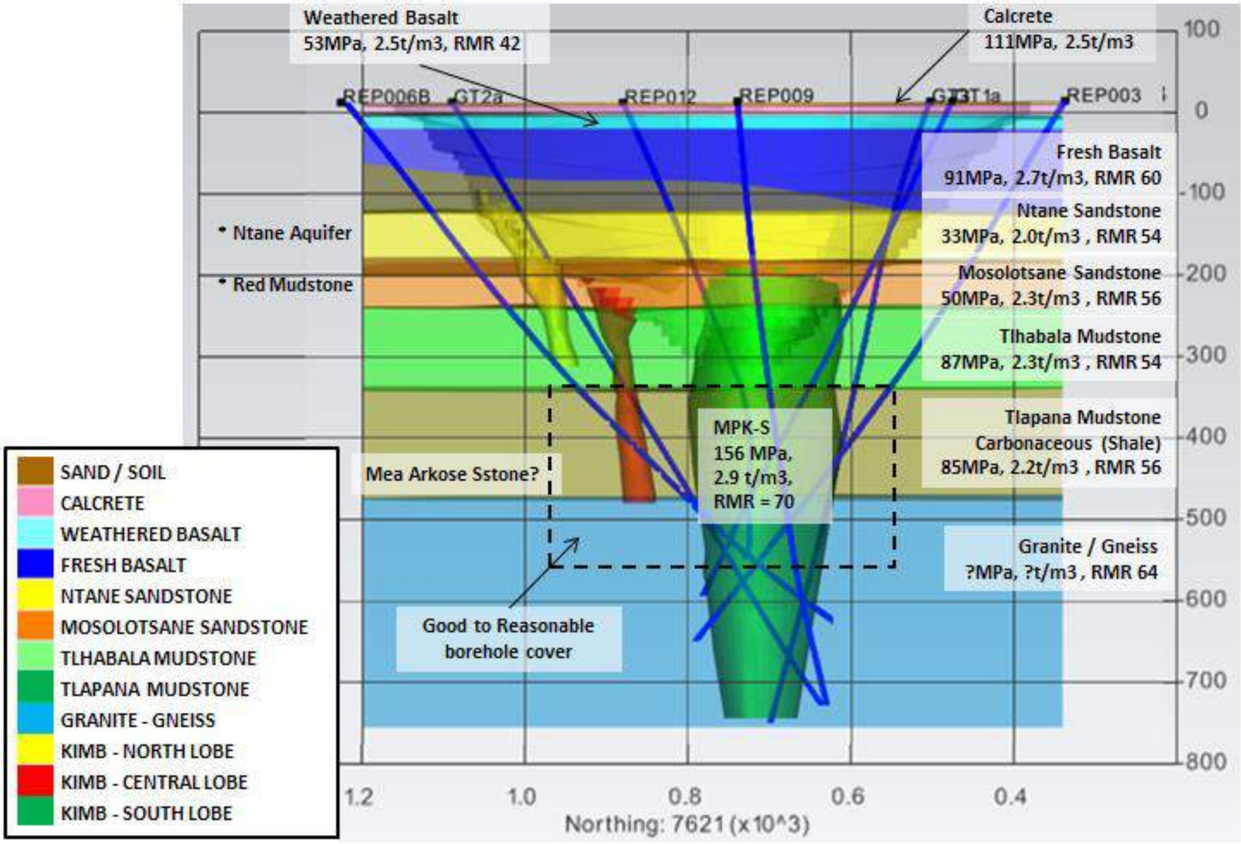


Figure 16.12 - North-south section through the geological model summarising known geotechnical parameters

16.1.4 Rock strength and known rockmass parameters

Rock strength and geotechnical parameters from past geotechnical reports are described below. Laboratory testing of samples from the 2016 / 2017 drilling are underway.

In Table 16.3 rock strengths of the various geological units are compared for the AK6 Karowe project and published data for Letlhakane mine. Note how much stronger the AK6 Mudstones have tested and also the general inference that the AK6 mudstone are less susceptible to weathering.

Table 16.3 - Rock Strength (UCS) - comparing AK6 Karowe and Letlhakane (after Ekkerd and Ruest, 2008)

Unit	UCS (MPa)	
	AK6	Letlhakane
Calcrete	111	-
Weathered Basalt	53	Not present / in model
Basalt	91	97
Ntane Sandstone	33	33
Moso Sandstone	49	Not present / in model
Tlabala Mudstone	87	34
Tlapana Mudstone	85	40

Observed kimberlite strength (UCS) and density (Table 16.4) is backed up by past field test work, however additional data collection and testing is required to understand the impact on caveability and thus on mining method selection.

Table 16.4 - Rock Strength (UCS) for AK6 Karowe (after Ekkerd and Ruest, 2008)

Unit	Count	Mean (MPa)	ST Dev (MPa)	Min (MPa)	Max (MPa)	COEF OF VARIATION	Elastic Modulus (GPa)	Density (t/m3)
MPK-S	205	156	30	58	216	20%	74	2.98
EMPK-S	36	103	35	37	201	34%	42	2.83

Unit	Count	UCS (MPa)				Elastic Modulus (GPa)	Density (t/m3)
		Mean	STD	Min	Max		
Calcrete	5	111	31	87	157	47.9	2.46
Weathered Basalt	15	53	27	18	94	17.8	2.46
Basalt	37	91	20	44	148	33.5	2.67
Ntane Sandstone	30	33	11	15	60	14.5	2.05
Moso Sandstone	20	49	21	24	97	18.8	2.29
Tlabala Mudstone	28	87	19	50	125	15.8	2.31
Tlapana Mudstone	9	85	39	25	135	13.5	2.24

All known geotechnical averages are summarised in Table 16.5. In summary, from the published geotechnical parameters the following is noted:

- High density, high UCS, high RMR, good joint surface conditions and relatively sparse joint spacing all contribute to concerns with respect to the natural caveability of the Karowe kimberlite.
- The competence of the Mudstone is unexpected based on regional data with significant impacts on development / support costs and the mining method.
- Additional drilling and geotechnical logging is therefore highly recommended.

Table 16.5 - Summary of known geotechnical parameters

Geological unit	BTS (MPa)	UCS (MPa)	TCS (5MPa)	TCS (10MPa)	TCS (15MPa)	Density (t/m^3)	RMR
Calcrete	10.4	111	-	-	-	2.46	-
Weathered Basalt	4.6	53	-	-	-	2.46	42
Fresh Basalt	8.4	91	117	141	182	2.67	60
Ntane Sandstone	2.0	33	75	105	149	2.05	54
Mosolotsane Sstn	2.9	49	118	-	-	2.29	56
Tlhabala Mudstone	7.9	87	130	110	188	2.31	56
Tlapana Mudstone	7.1	85	-	-	-	2.24	54
Granite / Gneiss	-	-	-	-	-	-	64
Kimberlite (MPK-S)	-	156	-	-	-	2.98	70

16.1.5 Conclusion with regards to the geotechnical database

Recent geotechnical drilling indicates that all country rock intersections correlate well with the 2007 model requiring only minor adjustments. Changes to the geological model having a serious impact on the project are therefore deemed to be low risk. In support, Ekkerd and Ruest (2008) place the Karowe 2007 geological model at a high confidence level of 80-90% (post feasibility design and construction) – although the assessment was not focused on the underground operations.

Confirmation of rock strengths following completion of current laboratory testing of the GT and the REP drill hole core samples will address the following important strength properties:

1. High density and UCS values of the south lobe kimberlite.
2. The Mosolotsane mudstones at Karowe are more resilient than observed regionally (higher durability index due to much lower smectite clay content).

16.1.6 Geotechnical mine design considerations

Choosing the appropriate mining method is a task deferred to the next stage of study and the geotechnical considerations are discussed briefly here.

- A level-by-level blast and retreat method (like SLC) has some risk due to the amount of country rock development that must take place in the mudstone units. Avoiding excessive development in mudstones may require a mining method or layouts which positions the main extraction development deeper and in the basement granite. Other than the potential better mudstone conditions at Karowe other options do exist to mitigate risk i.e. ring drives located within the kimberlite extremities.
- The high kimberlite strength, density and competence (RMR) may impact on caveability and therefore on the applicability of the Block Cave method. As with SLC above, options and methods do exist to mitigate the risk. Caving of the kimberlite can be ensured through artificial means either through “pre-conditioning” of the rock mass.
- During the transition from open pit to an underground operation, it is most likely that the best solution will be some initial retreat mining (SLC) of the higher lying underground areas followed by another or same mining method from within the competent basement rocks. The selected mining method should aim to minimise development within the red mudstones of the Mosolotsane formation and/or the mudstone formations below the Mosolotsane.

For the PEA, a decision was taken to design a sublevel cave (SLC) retreat mining method. The following mine design geotechnical parameters were used for this PEA:

- Declines 6 m wide x 6.3 m high.
- Production cross cuts (5 m wide x 5 m high) spaced at 13 m apart centre-to-centre (8 m wide pillars).
- Levels spaced 25 m apart floor to floor (20 m middling).

These parameters will be reviewed in the PFS.

16.1.7 Mine support

Detailed support design is a task deferred to the next stage of study (the PFS). Recommendations for the support have been used for the purposes of costing in this PEA and are based on the various rock types and shown in Table 16.6.

Table 16.6 – Geotechnical Support Summary

Host Rock	Prevalence	Support
Development tunnels in the red mudstone	Limited - only where access tunnels are required. 5% or less of the total development assumed	Ground support solution might require a NATM[1] support approach. ▪ Water sealing ahead of the face (grout injection) ▪ Anchors steel sets and voidfill ▪ Reinforced concrete foot wall 400 to 500mm thick.
Development tunnels in Sandstones (Not the Mudstones)	High - assumed 30-40% of total non-kimberlite development assumed.	Welded mesh and shotcrete
Development tunnels in the Mudstones	High - 30-40% of total non-kimberlite development assumed.	Similar to the red mudstone above:
Retreat Tunnels in Kimberlite	Moderate to high (depends on if SLC or BC option is used)	Welded mesh and shotcrete Rock Water Sealant up to face

16.1.8 Gap analysis

Table 16.7 summarises a GAP analysis on the geotechnical data for the Karowe project.

[1] *The New Austrian tunneling method (NATM), also known as sequential excavation method (SEM), is a method of modern tunnel design and construction.*

Table 16.7 - Karowe geotechnical Gap Analysis

Data set / Information requirement	Description of Gap Identified	Remedial Action
Rock strength testing	No information exists for the Granite / Gneiss unit	Lab testing is underway and results will be available for use in the PFS.
Rock strength testing	Last updated in 2008. More applicable to the shallower lying areas.	Lab testing is underway and results will be available for use in the PFS.
Geotechnical drilling and coverage	There is a gap in the current coverage whereby the west orientation (270° trend) is not represented.	Further drilling recommended. This will likely take place during the PFS and results will be used to update reports and models for the FS.
Geotechnical drilling and coverage	Better drill hole coverage is required in the country rock for mining depths below 550m below surface.	Further drilling is recommended. This will likely take place during the PFS and results will only be used to update reports and models for the FS.
Geological Model	The Geological model was last updated in 2013 and is still based on the 2007 interpretation.	An update is being prepared and will be available for the PFS.
Geotechnical model / database	All historic geotechnical logs are to be sourced and investigated for relevant insights and inclusion into an updated geotechnical model.	A detailed analysis of the 2016 / 2017 drill hole geotechnical logs is currently underway. Results should inform the PFS.
Geohydrology	Lack of geohydrological information at depth below current LOM of the open pit.	Refer to Geohydrology section.

16.1.9 Forward work plan

A further geotechnical investigation, based on a drilling programme is recommended. The programme involves significant additional geotechnical drilling to take place during 2018 and as summarised in Table 16.8 below. A 3D view of the planned drilling is shown in Figure 16.13. The drilling will increase the confidence of the data set to comply with feasibility study level of accuracy.

It is unlikely that any further site drilling or logging will take place in time with results ready for use in the PFS. This information is however critical to address the various data gaps identified. The additional drilling and site work will most likely run in parallel with both the PFS and the FS. The additional geotechnical information should be analysed in a timely manner and ensure that the PFS and FS designs are updated accordingly.

Table 16.8 - Karowe geotechnical FWP drilling

No. of Drill holes	Cumulative length (m)	Purpose	Priority
8	4,340	Sub vertical holes from surface to the bottom of the UG targeting the host rockmass where waste development and/or rim development will take place. These holes are critical to better understanding of the extensive carbonaceous shales and other zones of weakness. Full time site geotechnical presence and immediate out-the-hole strength testing and logging will be required.	1
1	634	Sub vertical in-pit hole from surface to the bottom of the UG targeting the centre of the South kimberlite lobe. To intersect the lobe in its vertical extent from top to the bottom of planned UG mining.	1
8	6,820	Sub vertical holes from surface to the bottom of the UG targeting both the waste rock and the kimberlite where resolution was poor for the PEA assessment.	2
4	2,880	Inclined holes from surface to the bottom of the UG mining through the planned spiral decline access locations. Two holes for each the north and south decline access ways.	3
17 (est.)	3,000	Shallow inclined holes targeting the portal location (for portal design) and targeting the lateral extent of the planned decline ramps.	3

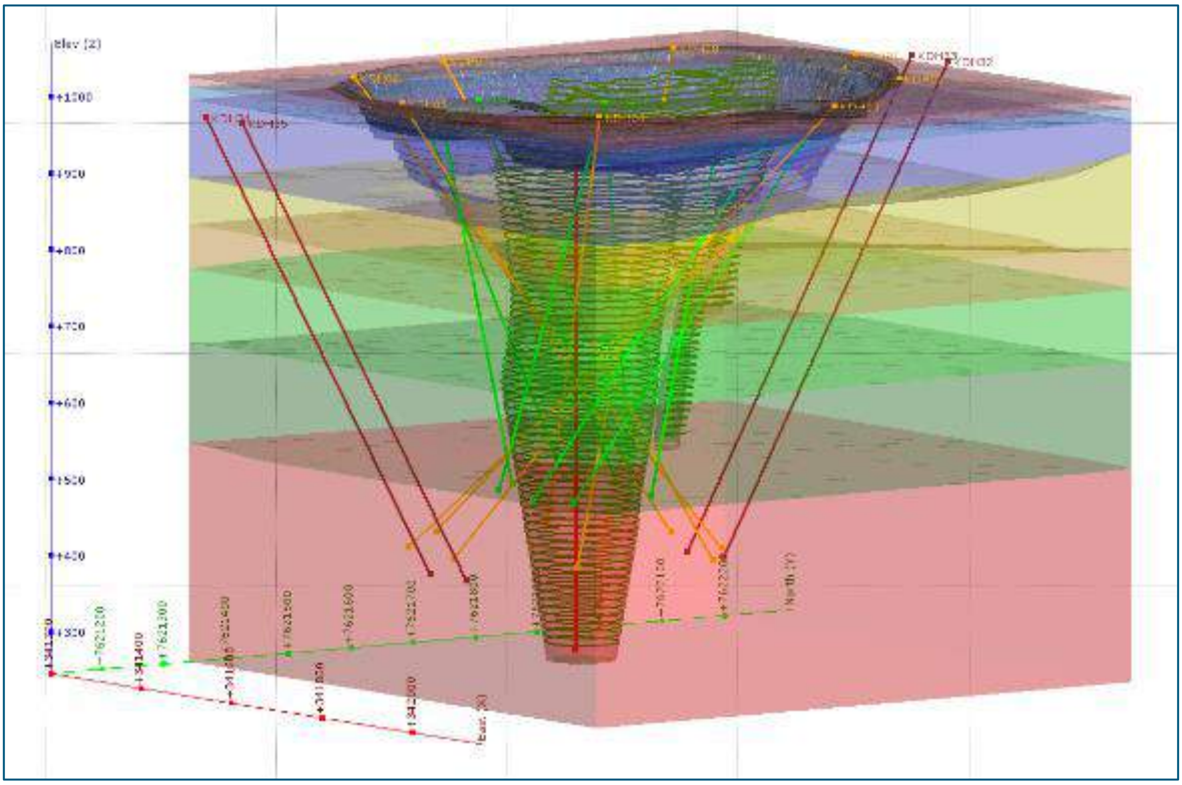


Figure 16.13 - Planned geotechnical drilling as part of the FWP

16.2 Hydrogeology

16.2.1 Regional and local hydrogeology

The Hydrogeology of the area is well known and the main aquifers have been supplying adjacent mines Orapa, Letlhakane and Damtshaa (OLD) with over 12 Mm^3/yr of water for nearly 40 years. The dewatering strategy for Orapa and Letlhakane open pits has been effective to circa 350 mbs.

The geology and general hydrostratigraphic units of the Karowe area are from surface down:

- Kalahari sand and calcrete.
- Stormberg Basalt.
- Ntane sandstone.
- Mosolotsane red mudstones and sandstone.
- Tlhabala mudstone.
- Tlapana carbonaceous mudstone.
- Mea Arkose siltstone and sandstone.
- Basement granite (weathered upper zone and unaltered).

Figure 16.14 shows the general regional stratigraphic correlation illustrating that Karowe hydrogeology is very similar to Letlhakane and Orapa.

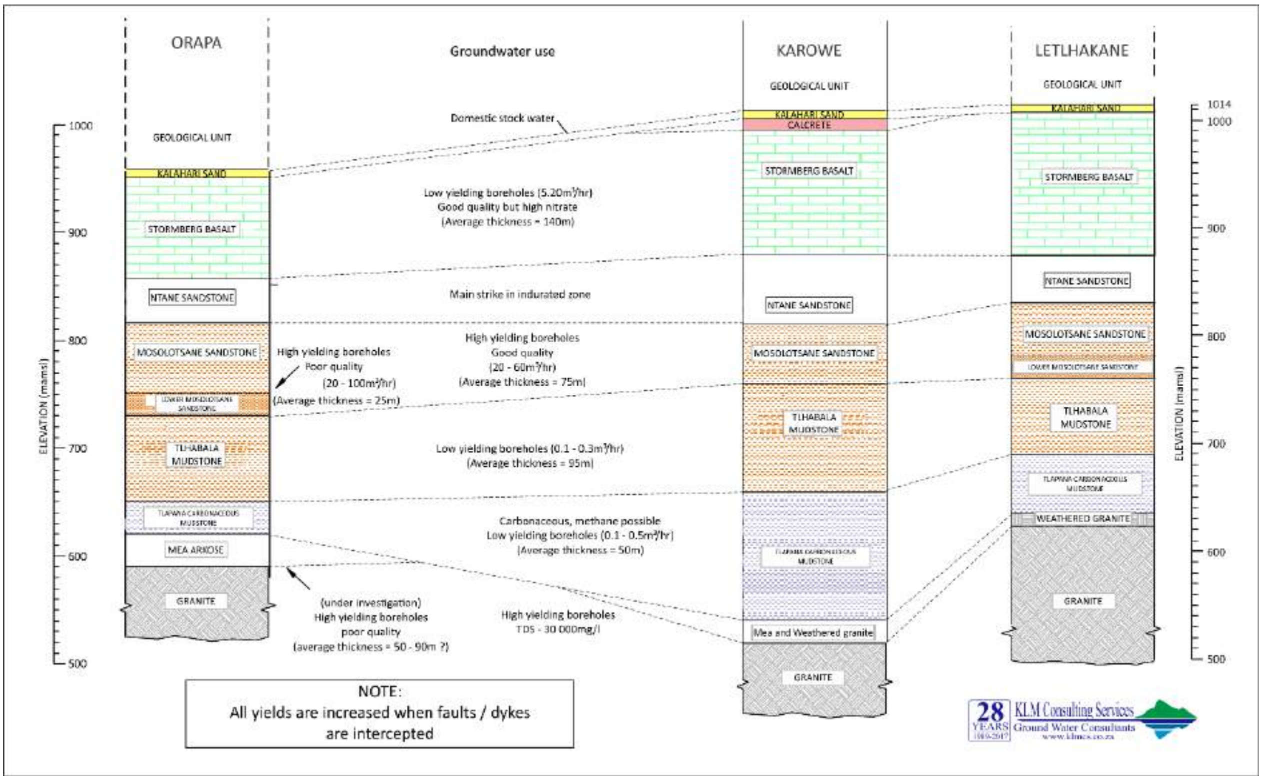


Figure 16.14 - Cross section of stratigraphic correlation between Orapa, Karowe and Letlhakane mines

Table 16.9 summarises the aquifer hydraulic properties for the upper geological units as collected by KLMCS 2007 and WSB (2005, 2007) from packer testing by KLMCS (2007) and test pumping of 16 dewatering boreholes by KLMCS (2007 and 2011). Packer testing by KLMCS enabled isolating the different aquifer hydraulic characteristics, whereas the test pumping data from the 16 dewatering boreholes gave a hybrid of the two Ntane and Mosolotsane aquifers.

Table 16.9 - Aquifer hydraulic properties derived from test pumping and packer tests

Source	Type of Test	Aquifer Hydraulic Properties						
		Hydraulic conductivity, K m/day				Storage coefficient/Specific yield, S/Sy (dimensionless)		
		Basalt	Ntane	Mosolotsane	Tlhabala	Basalt	Ntane	Mosolotsane
KLMCS, 2007 AK6 dewatering prefeasibility study	Packer Testing (core holes H004 and H007)	0.0001 0.01	0.005 0.5	0.1	<0.0001			
	Test pumping (Pumping boreholes H002 and H005)		0.07				$1x10^{-4}/3x10^{-2}$	
WSB Wellfield 7 study (only those boreholes close to AK6 have been included)	Test pumping Z12239 Z12299 Z12517 Z12520 Z12521		 0.16-0.21 0.17-0.24 0.25 0.08 0.05				 $1.5x10^{-4}/2x10^{-4}$ $2x10^{-5}/$ $5x10^{-4}/8x10^{-4}$ $2x10^{-5}/4.5x10^{-3}$ $1x10^{-4}/8x10^{-4}$	
WSB, 2007 AK6 wellfield study drilling and testing of 4 boreholes (Z12795-Z12798)			0.1-0.2				$1x10^{-4}/2.5x10^{-3}$	
WSB, 2007	Compilation of test pumping data from 247 bhs in Orapa/LM area	0.05	0.1-0.4	0.016		$1x10^{-5}/1x10^{-3}$	$5x10^{-5}/1x10^{-3}$	$1x10^{-5}/1x10^{-3}$
KLMCS, 2011	Test pumping of 16 dewatering boreholes	-	0.1-0.2 0.35-0.5(along structures)				$3x10^{-5}-1x10^{-4}/5x10-3$	

The aquifers are compartmented by hydrogeological boundaries comprising:

- The piezometry shows a SE-NW to SSE-NNW groundwater flow direction.
- Inflow from the SE/SSE defines the influx boundary into the AK6 area.
- Outflow to the N and NNW define the out flux boundary.
- The East and West are no-flow boundaries as groundwater flow is parallel to these boundaries.
- Internal partial barrier boundaries are defined by NW-SE faults and dykes and NNW-SSE faults.

Summary of hydrogeology characteristics

The regional groundwater flow is driven by recharge via the outcropping Ntane sandstone on the escarpment at Serowe. The regional SE-NW flow is by piston flow. The net local groundwater balance is nil recharge and no change in the storage term of the groundwater system, reflected by relatively unchanging water levels. Travel time and residence time is long (hundreds of years), resulting in dissolution of minerals from the host rock and saline groundwater as recorded from the groundwater samples from AK6 boreholes.

The hydrogeological units for Karowe mine have been summarised in Table 16.10 showing the hydrostratigraphic unit thickness, depth below ground, hydraulic conductivity (both horizontal and vertical) and storage characteristics of each hydrostratigraphic unit.

Table 16.10 – Summary of Hydrogeological units and hydraulic values

Unit	Thickness (m)	Depth (mbs)	Horizontal Hydraulic conductivity (m/d)	Vertical Hydraulic conductivity (m/d)	Specific storage	Specific yield	Comments
Basalt	130	130	0.05	0.01	$2x10^{-6}$	$1x10^{-3}$	Low K and S due to primary porosity.
Ntane sandstone	70	200	0.15	0.15	$3x10^{-5}$	$5x10^{-3}$ - $2x10^{-2}$	High K and S due to primary and secondary porosity
Upper Mosolotsane	40	240	0.024	0.024	$2x10^{-6}$	$5x10^{-3}$	Low permeability due to clay/mudstone
Lower Mosolotsane	16	256	0.1	0.1	$2x10^{-6}$	$5x10^{-3}$	High permeability due to coarse sandstone units
Tlhabala mudstone	90	346	0.0005	0.0005	$3x10^{-5}$	$1x10^{-3}$	Very low permeability due to mudstone
Tlapana mudstone	45	391	0.0005	0.0005	$3x10^{-5}$	$1x10^{-3}$	Very low permeability due to carbonaceous mudstone
Upper granite	100	491	0.05	0.05	$2x10^{-6}$	$1x10^{-3}$	Moderate permeability due to fracturing of the upper granite and weathering

Unit	Thickness (m)	Depth (mbs)	Horizontal Hydraulic conductivity (m/d)	Vertical Hydraulic conductivity (m/d)	Specific storage	Specific yield	Comments
Lower granite	300	791	0.005	0.005	$2x10^{-6}$	$1x10^{-3}$	Low permeability due to predominantly massive un-fractured rock
Upper kimberlite	130	130	0.02	0.02	$5x10^{-6}$	$1x10^{-4}$	Moderate permeability due to weathering/fracturing and pressure release.
Middle kimberlite	70	200	0.005	0.005	$5x10^{-6}$	$1x10^{-4}$	Low permeability due to pressure release
Lower kimberlite	>791	>791	0.001	0.001	$5x10^{-6}$	$1x10^{-4}$	Very low permeability due to massive un-fractured rock
NW-SE Dykes and NNW-SSE faults	>791 Width- 50m-500m	>791	0.00001	0.01	-	-	Partial barriers defined by dolerite dykes and faults striking NNW and Nw.
High permeability structures	20m-100m	>791	0.35-0.5	0.1	$3x10^{-5}$	$2x10^{-2}$	Highly permeable structures mapped from correlation of transmissivity and yields with structural maps

Two recent numerical ground water models were completed for Karowe mine. The first by KLMCS using MODFLOW and the second by Itasca in 2016 using MINEDEW.

The results of the KLMCS model were used to design the initial water supply and dewatering strategy for Karowe mine in 2011. The Itasca model has been used to recommend upgrades to the 2017 dewatering strategy. A new numerical model is in progress using FEFLOW and this will be held on the mine for use in operational planning and implementation. The new model will be used to develop the specific dewatering strategy for the underground mine.

Groundwater Recharge

Recharge to ground water for the area has consistently exceeded predictions. This is due to the enormous extent of the Ntane and Mosolotsane aquifers which stretch as far south as Lesotho, east to Bulawayo in Zimbabwe, west to central Namibia and north to the Democratic Republic of the Congo.

All water chemistry samples collected during drilling and test pumping from the initial 16 dewatering boreholes exhibit a Na-Cl water type. Previous hydrochemical studies in the area by WSB (2005, 2007) also show a dominant Sodium Chloride (Na-Cl) water type with Total Dissolved Solids (TDS) of >2000mg/l. It is possible, from the dominance of Na and Cl in the water, that the bulk of the groundwater was recharged during the last pluvial period 5000 years-10000 years ago.

16.2.2 Dewatering of the current open pit

Figure 16.15 shows the layout of the 2017 pit dewatering boreholes and geological structures.

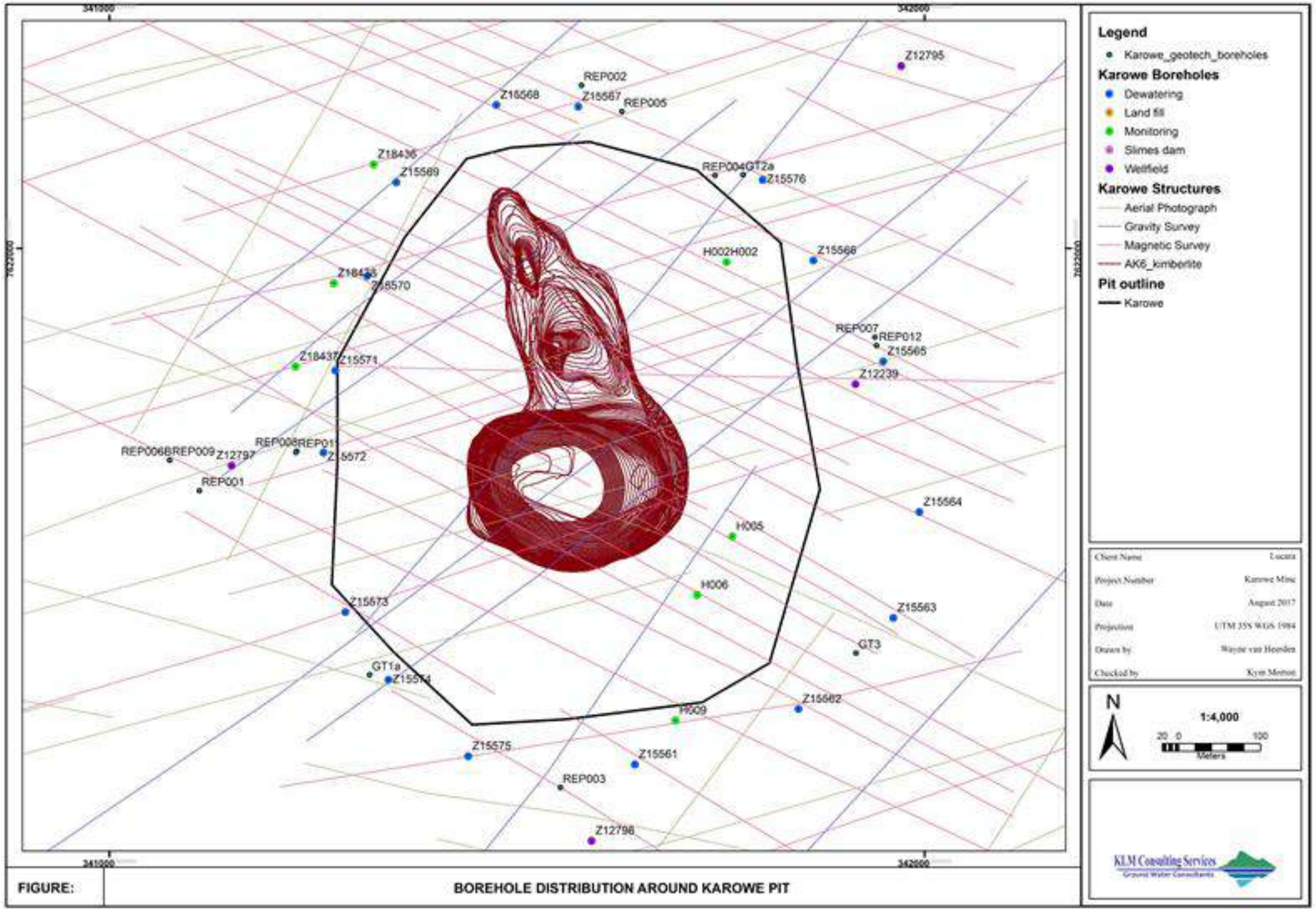


Figure 16.15 - Layout of Pit perimeter dewatering boreholes and geological structures

As of November 2017 the mine operates fifteen pit perimeter dewatering boreholes; twelve electric powered and three diesel. The 15 dewatering boreholes have a combined yield of 243 m^3/hr. Six (6) wellfield boreholes supply a combined yield of 90 m^3/hr and are all electric powered. The six wellfield boreholes are only pumped to augment water supply for mining operations. Table 16.11 shows a summary of the November 2017 status of all boreholes in the mine.

Table 16.11 – November 2017 Borehole Status

	BH ID	Official No.	Actual Rate (m^3/hr)	Status (Running/ Not Running)
Dewatering Boreholes - Electric Powered	003	Z15564	10.0	Operational
	004	Z15561	12.0	Operational
	005	Z15566	5.0	Operational
	008	Z15572	24.5	Operational
	009	Z15571	24.8	Operational
	010	Z15568	30.0	Operational
	011	Z15575	11.0	Operational
	012	Z15562	19.0	Operational
	013	Z15565	10.0	Operational
	014	Z15567	11.0	Operational
	015	Z15569	20.5	Operational
	016	Z15570	25.0	Operational
Dewatering Boreholes - Diesel	WS1	Z12795	11.0	Operational
	WS3	Z12797	12.0	Operational
	WS5	Z12299	18.0	Operational
Wellfield Boreholes - Electric Powered	022	Z18430	11.0	Operational
	023	Z18431	18.0	Operational
	024	Z18432	12.0	Operational
	025	Z18433	11.0	Operational
	026	Z18434	11.8	Operational
	027	Z18435	30.0	Operational

The current dewatering target aims to achieve a daily volume pumped of 185 m^3/hr, based on the last ground water model, which had assumed 16 pumping boreholes being operational at any given time. At this rate, according to the Itasca predictions, the water levels are expected to be decreased to approximately 800 mamsl by 2021.

Figure 16.16 shows the water levels in the monitoring boreholes and point piezometers as a graph with the planned pit bottom over time. It shows the water level is close to pit bottom and has not been lowering since 2012. The required dewatering rate is 25 m/yr therefore Karowe mine has embarked on a fast tracked strategy to achieve the required dewatering rate.

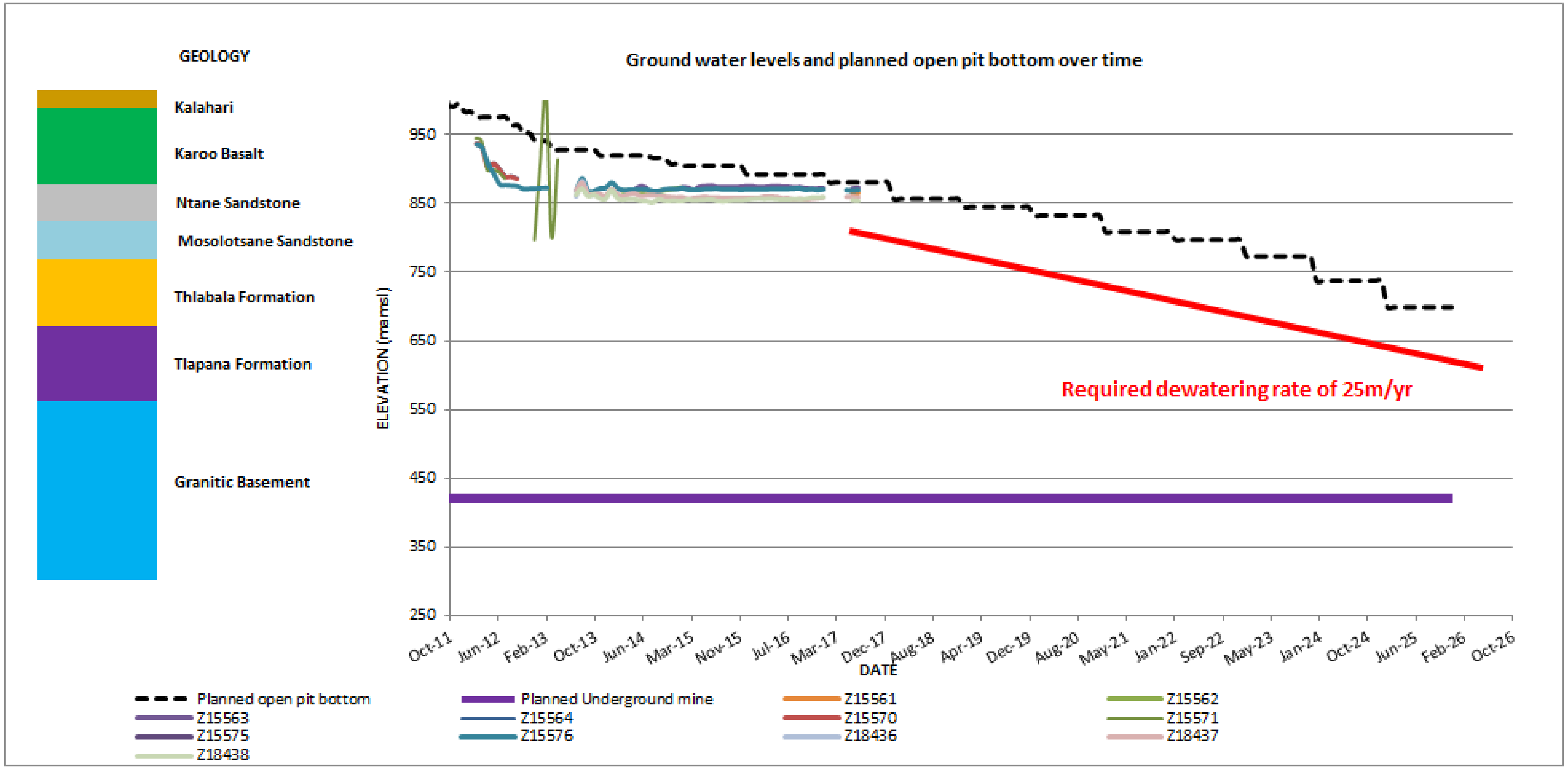


Figure 16.16 - Water levels and planned pit bottom over time

16.2.3 Dewatering to support open pit and underground

The proposed underground mine will assist in dewatering the open pit and a dewatered open pit is a prerequisite for safe underground mining. Accurate dewatering comprises:

- Storm Water Control.
- Sump pumping.
- Pit perimeter pumping of Ntane and Mosolotsane aquifers.
- Pit perimeter pumping of Mea aquifer, granite contact and structures in granite.
- Sub horizontal drain holes (snake holes).
- In-Pit dewatering boreholes.
- Underground gallery.

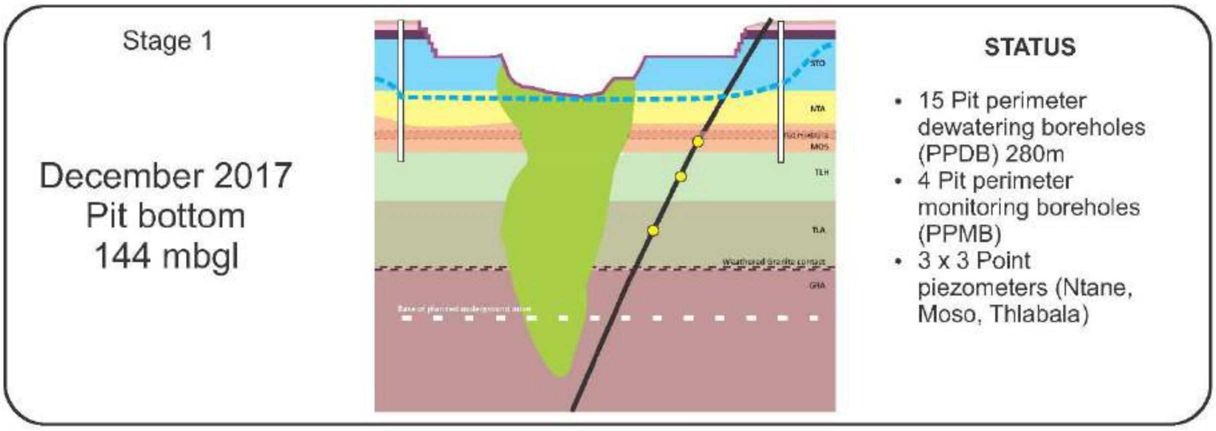
All of the above are in progress at Karowe. The feasibility of installing a dewatering ring tunnel around the pipe at the granite contact and/or the use of early access tunnels will be evaluated during the PFS. This will be more effective than pit perimeter wells, however success is dependent on the timing of the start of underground development preceded by ramp development and/or shaft sinking.

The measures outlined above will be utilised to achieve targeted drawdowns (∆h) in the water levels and pore pressures for each geological unit for specific sectors of the mine. Of specific relevance is the pore pressure in the mudstone unit within and below the Mosolotsane sandstones. These formations are known to be weak and the design slope angle will take into account the amount of depressurizing that can be created by advanced dewatering.

Traditional diamond mine dewatering (Kimberley, De Beers', Koffiefontein, Bultfontein, Du Toitspan and Wesselton mines) used the installation of dewatering tunnels to intercept the country rock inflows around the open pit and in advance of underground mining (Morton 2008).

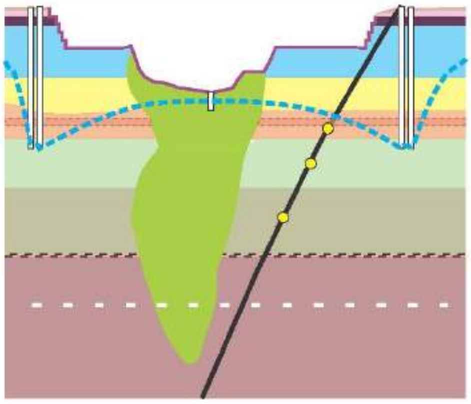
Gravity drainage is more effective than multiple pumping boreholes but the initial capital expenditure is high. Early access tunnels for either block cave or kimberlite sampling can offset the capital expenditure. Both these options will be evaluated at PFS stage. All mining methods benefit by dewatering in advance of mining. Mine design with an integrated dewatering design is recommended.

Figure 16.17 shows the schematic diagrams illustrating the proposed steps in dewatering the pit and then the underground over the life of mine. The new numerical model will confirm the number of boreholes required, the timing and use of an underground ring tunnel and the volumes of water to be removed by specific target dates.



Stage 2

June 2018
Pit bottom
168 mbgl

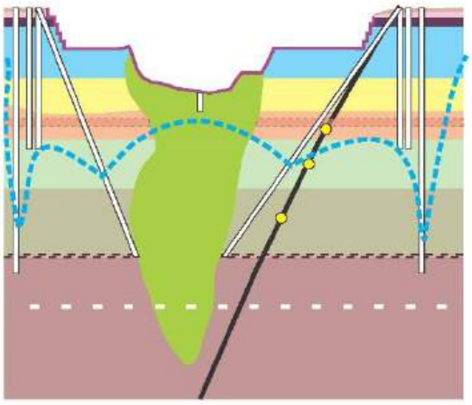


STATUS

- 15 PPDB to 280m
- 4 New design boreholes (NDBH) to 280m
- 4 PPMB
- 5 In pit monitoring boreholes (IPMB)
- 3 x 3 Point piezometers (Ntane, Moso, Thlabala)

Stage 3

September 2018
Pit bottom
168 mbgl

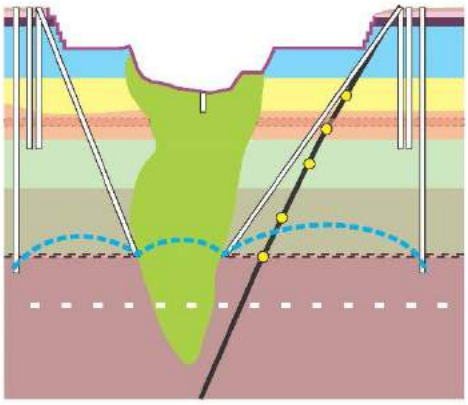


STATUS

- 15 PPDB to 280m
- 4 NDBH to 280m
- 16 PPDB to 410m
- 4 PPMB
- 5 IPMB
- 3 x 3 Point piezometers
- 2 x 12inch angled deep dewatering boreholes (ADDB)
- 6 x Vertical deep dewatering boreholes (VDDB)

Stage 4

December 2018
Pit bottom
168 mbgl



STATUS

- 15 PPDB to 280m
- 4 NDBH to 280m
- 16 PPDB to 410m
- 4 PPMB
- 5 IPMB
- 2 x 12inch ADDB
- 6 x VDDB
- 3 x 5 Point piezometers (Ntane, Moso, Thlabala, Tlapana, Granite/MEA)

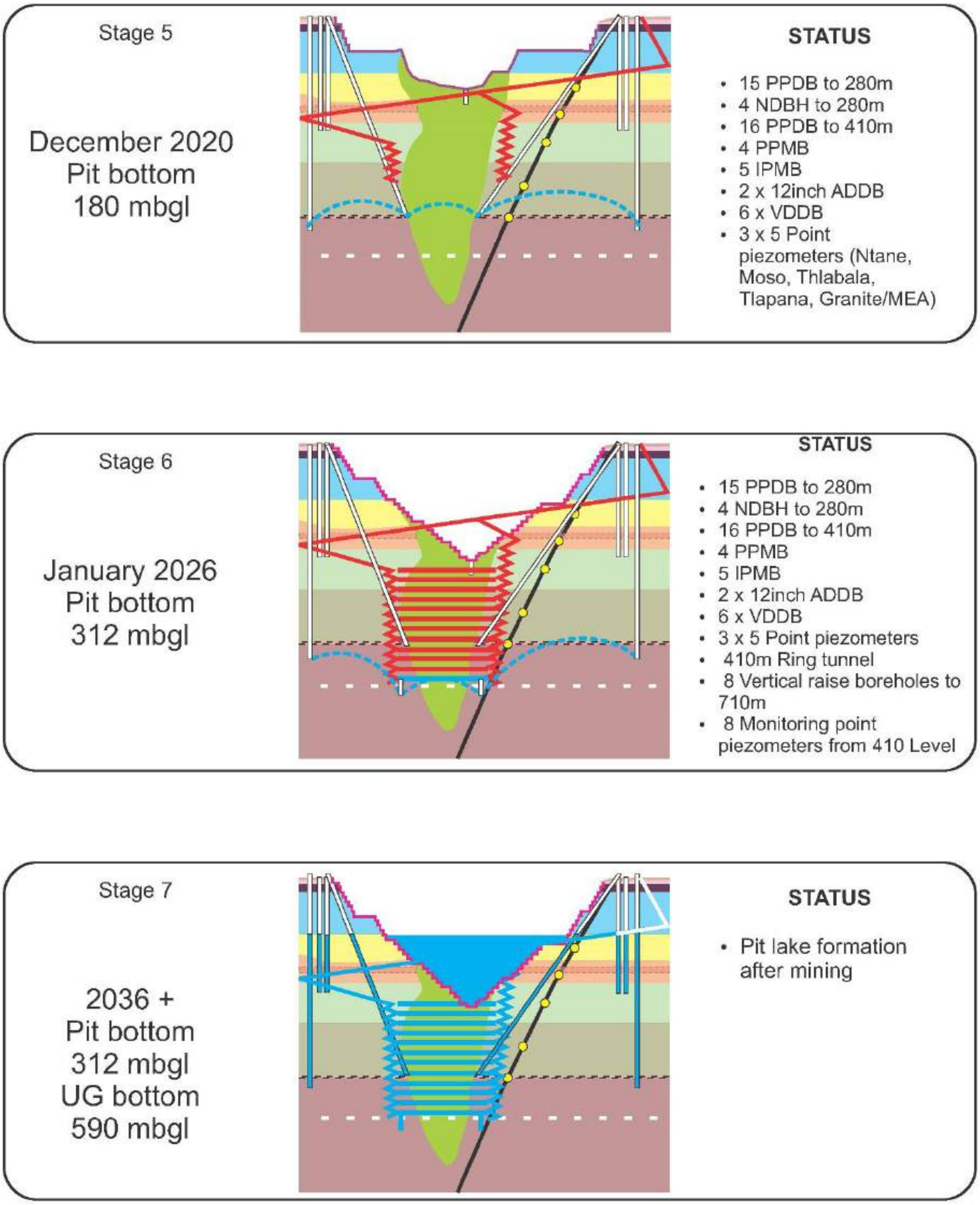


Figure 16.17 - Schematic diagrams illustrating the Stages of Dewatering

Proposed stages of dewatering

Stage 1 November 2017

- 15 active pit perimeter dewatering boreholes (PPDB) to c280 mbs on the pit perimeter.
- Four monitoring water level monitoring boreholes within the perimeter outline (PPMB).
- Three point piezometers measuring water pressure in angled core holes below the pit slopes (Ntane, Mosolotsane and Tlhabala hydrogeological units).
- Sump pumping when required.
- The water levels are not declining and are 10 - 15 m below pit bottom. Accelerated dewatering is planned.

Stage 2 Planned for June 2018 (timing dependent on Jan 2018 model results)

Stage 1 plus:

- 5 in pit water level monitoring boreholes.
- 4 New design boreholes (NDBH) to 280m

Stage 3 September 2018 (timing dependent on Jan 2018 model results)

Stage 2 plus:

- 2 x 12 inch diameter angled deep dewatering boreholes targeting kimberlite contact.
- 6 vertical deep dewatering boreholes to 410 mbs (to base of weathered granite). (Number dependent on model results).

Stage 4 December 2018 (timing dependent on Jan 2018 model results)

Stage 3 plus:

- 3 x 5 point piezometer pressure monitoring probes in 3 angled geotechnical core holes below pit slopes and in Ntane, Mosolotsane, Tlhabala, Tlapana and Mea/granite contact hydrogeological zones.
- Drain holes from within pit to mudstone horizons.

Stage 5 December 2020

Stage 4 plus:

- Installation of ramps, decline and access to SLC infrastructure. Additional dewatering of infrastructure using flyte pumps and cover drilling/drain holes.

Stage 6 January 2026 Active underground mining

Stage 5 plus:

- Ring tunnel at circa 410 mbs.
- Vertical raise boreholes from ring tunnel to circa 710 mbs.
- 8 underground pore pressure monitoring points drilled from underground access at 410 mbs.

Stage 7 2036 + end of mine

- All pumping stopped and pit flooded to close to pre mining level (rate of rebound will be modelled in PFS).

The new numerical model will confirm the number of boreholes required for the open pit dewatering. The model will also be used in the PFS to estimate the timing and use of an underground ring tunnel and raise boreholes. The volumes of water to be removed by specific target dates will be estimated. Water rebound and environmental impact will be simulated.

16.2.4 Estimate of inflows to the proposed underground mine

Figure 16.18 shows the geology, planned pit shells and possible final underground depth.

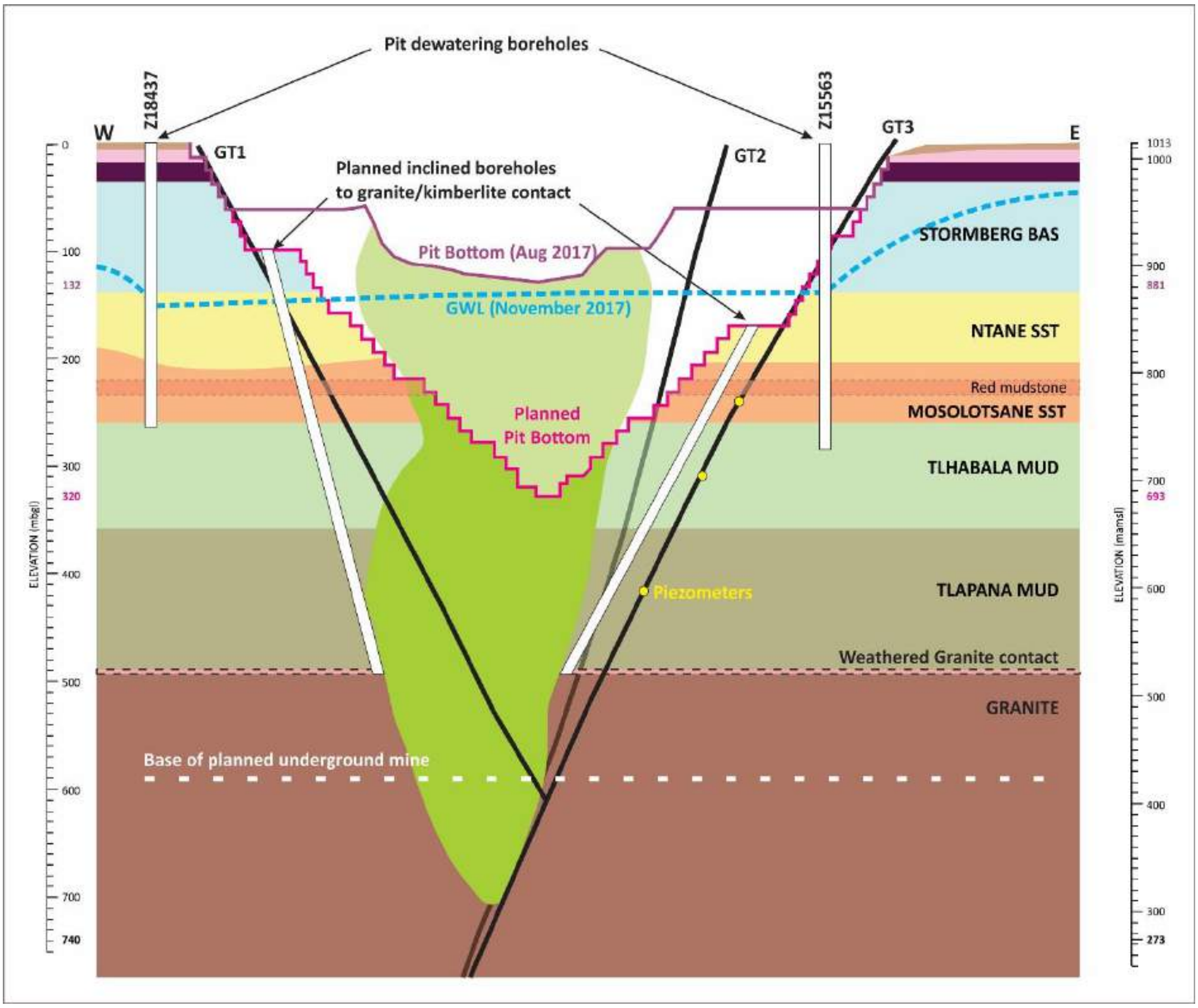


Figure 16.18 - Karowe cross section facing north showing July water levels, planned 2018 replacement boreholes and depth of underground mine

Inflows have been calculated for the open pit using the following models:

2006 KLM Consulting Services Conceptual Model: A preliminary hydrogeological conceptual model was developed and used in the pre-feasibility studies (KLMCS 2006). Data was obtained from nearby Debswana Mines.

2007 Hydrologic Consultants Inc. (HCI) Groundwater Model: A preliminary groundwater flow model was developed and used in the pre-feasibility study (HCI 2007a, 2007b). A preliminary dewatering system was simulated with the available hydrogeological data. The HCI 2007 model was used in the 2015 Itasca model.

2007 Dewatering Pre-Feasibility Study: Six pumping boreholes were constructed and drilled. Hydraulic testing and monitoring were conducted on these boreholes (KLMCS 2007).

2007 Structure Reports: Barnett (2006, 2007) performed a geophysical and geotechnical analysis and determined that six dominant joint sets exist in the Karowe pit.

2010 and 2011 Karowe Study: Over a two-year period, KLMCS (2010, 2011) conducted a hydrogeological evaluation of the Karowe pit, a geophysical survey, a water-management evaluation, and a water-quality assessment, and developed a groundwater monitoring report.

2011 and 2014 Groundwater Model Reports: KLMCS developed two models, with a local- scale model (KLMCS 2011) being used for pit dewatering and a regional-scale model (KLMCS 2014) being used for regional water supply.

2013 and 2014 Groundwater Monitoring Reports: Boteti Mining has produced yearly groundwater monitoring reports at Karowe beginning in 2013 (Boteti Mining 2013, 2014). Each report details the dewatering rates, water-supply abstraction rates, and the effects on the local groundwater system. Integration of recent yearly reports is underway and can then be used to calibrate and predict dewatering effectiveness for the open pit.

In 2015 the Itasca Model simulated the following predictive scenarios:

Scenario 1: This scenario only simulated the 2014 dewatering system. The 10 active perimeter dewatering boreholes continued pumping at the December 2014 dewatering rates assuming 100% utilization rates and efficiencies.

Scenario 2: This scenario simulated an additional four in-pit dewatering boreholes. The 10 active dewatering boreholes were pumped at the December 2014 rates until the end of model simulation. Two additional dewatering boreholes are added in mid-2020, another in mid-2021, and one more in-pit dewatering borehole at the beginning of 2023. All dewatering boreholes were assumed to have 100% utilization rates and efficiencies.

Conclusions from the 2015 Itasca Model:

Sixteen dewatering boreholes are sufficient to decrease the groundwater levels to approximately 800 mamsl (213 mbs) if all boreholes are efficient and fully utilised. Once the pit-bottom elevation is excavated below 800 mamsl, Residual Passive Inflow (RPI) will occur if no additional in-pit dewatering boreholes are installed. Estimated RPI reaches a peak of 417 m³/hr in 2023 if only the existing 16 dewatering boreholes are used.

Adding four in-pit dewatering boreholes, in addition to the existing 16 boreholes, can maintain dry pit conditions for the Life of Mine (LOM) to 360 m depth. The contact zone was targeted for new dewatering boreholes based on the assumption that it has relatively high K values. Total dewatering rates ranging between 175 and 210 m³/hr will be sufficient to maintain dry pit conditions to 360 m depth.

The deepest existing pumping boreholes only penetrate to between 283 m and 295 m and are too shallow for dewatering to below 280 m (733 mamsl). The Itasca model has indicated that the pit will have additional inflows from a pit bottom of 213 m and that a total of 20 deeper boreholes pumping at 100% efficiency and utilization are required to dewater the pit to 360 m depth. The current dewatering design is not capable of dewatering the planned open pit, therefore additional boreholes are planned for 2018.

An updated numerical model will estimate the inflows to the proposed deeper open pit and the proposed underground mine. Information from the upgraded monitoring network will be used to calibrate the revised numerical model.

Inflows to a mine over time are not linear. As each cut, new development or drop down is initiated flows increase then level out. Using the 2017 inflows to a pit depth of 881 mamsl (132 mbs) and an analytical solution the estimated maximum inflows for the open pit and underground mine are shown in Table 16.12 below.

Table 16.12 – Estimated Inflows

Depth	Estimated worst case inflow	Comment
Open pit to 320 mbs (590 mamsl)	417 m³/hr to 213 mbs (800 mamsl) then plus 175 to 210 m³/hr to 320 mbs (590 mamsl) Total of 600 to 627 m³/hr Total 5 Mm³/yr	Assumes efficient pumping and 100% utilization (Source: Itasca 2015 model). Requires deepening and new design for borehole construction.
Underground to 740 mbs	Additional 200 m³/hr (1.75 Mm³/yr)	Depends on extent of Mea aquifer, regional structures and /or fractured and weathered granite.

Total maximum inflow for the open pit and underground mine will be between 3.2 Mm³/yr and 6.7 Mm³yr depending on speed of mining and how much water is extracted up-gradient.

The revised structural model will be overlain with the borehole yields to locate high yielding water bearing structures. These can then be intercepted with pumping boreholes up-gradient of the mine, or where they intersect the kimberlite contact. Average yields for long term pumping boreholes are 20 m³/hr. There are sectors of the mine that have lower yielding boreholes. The distribution of the varying borehole yields will be evaluated as part of the upgrading of the open pit dewatering design. To obtain a more accurate estimate of total inflows the conceptual model requires the addition of hydraulic parameters for the units below the Mosolotsane followed by numerical modelling.

The probable impact of the mine on regional ground water levels will be around 5 km radius from the mine lease perimeter (KLMCS 2007) however this will be more accurately predicted using an updated numerical model and 2018 regional borehole census.

16.2.5 Recommendations

Recommendations for the drilling and testing of lower units

The upper aquifer units are well known. Drilling and testing of the lower units is required to confirm the extent and hydraulic characteristics of the lower units. The best method to determine location of ground water, hydraulic values and rates of inflow is the installation of large diameter boreholes and aquifer testing down to the granite using pumps and observation boreholes.

The following tests are recommended:

- Core drilling of the Tlhabala, Tlapana, Mea and granite in combination with the geotechnical investigation. Three coreholes located in the SE, SW and NW to fill in the gaps between the GT coreholes. These should be equipped with point piezometers in the Tlhabala, Tlapana and Mea or granite units.
- Packer testing of three of the geotechnical core holes to obtain hydraulic conductivity of the Tlhabala and Tlapana formations in discrete sections of the core holes.
- Drilling of four deep, large diameter percussion boreholes into the Mea or granite contact in four quadrants of the mine area at the pit perimeter NE, NW, SE M SW.
- Step tests, 48 hr constant discharge and recovery pumping tests. Accurate measurements of dewatering pumping and water levels will be required at the same time.
- Hydrochemistry sampling of the Tlapana, Mea and granite will be done at the same time as the drilling and pump testing

Optimisation of the drilling and testing programme will be done to meet geotechnical, hydrogeological and environmental needs.

Recommendations for the monitoring network

The monitoring network will comprise:

- Regional water levels (boreholes at the lease area perimeter or farmer's boreholes).
- Monitoring boreholes within the lease area and wellfield.
- Point piezometers in the Tlhabala, Tlapana and Mea/granite hydrogeological units.(3 x 3 coreholes twined with geotechnical use)
- Temporary, sacrificial, shallow (30 m) water level monitoring boreholes within the pit (5 replaced as required).
- Pumping rates and volumes from boreholes and sumps.
- Rainfall gauges.
- Monitoring points below seepage sites (tailings and waste rock dumps).

Three GT coreholes have been equipped with point piezometers to measure pressure in the Ntane, Mosolotsane and Tlhabala units.

All data will be readily available through the data management system planned by the mine and accessible to the project team. Water level targets will be set for each mining sector.

Recommendations for the current pit dewatering design

Karowe mine plan to drill four new 280 m pumping boreholes to upgrade dewatering of the pit, they will be sited within the circumference of the pit perimeter ring of existing dewatering boreholes and outside of the zone of relaxation. Two additional boreholes are planned for within the pit to intercept the kimberlite contact with the granite. Figure 16.19 shows the planned locations.

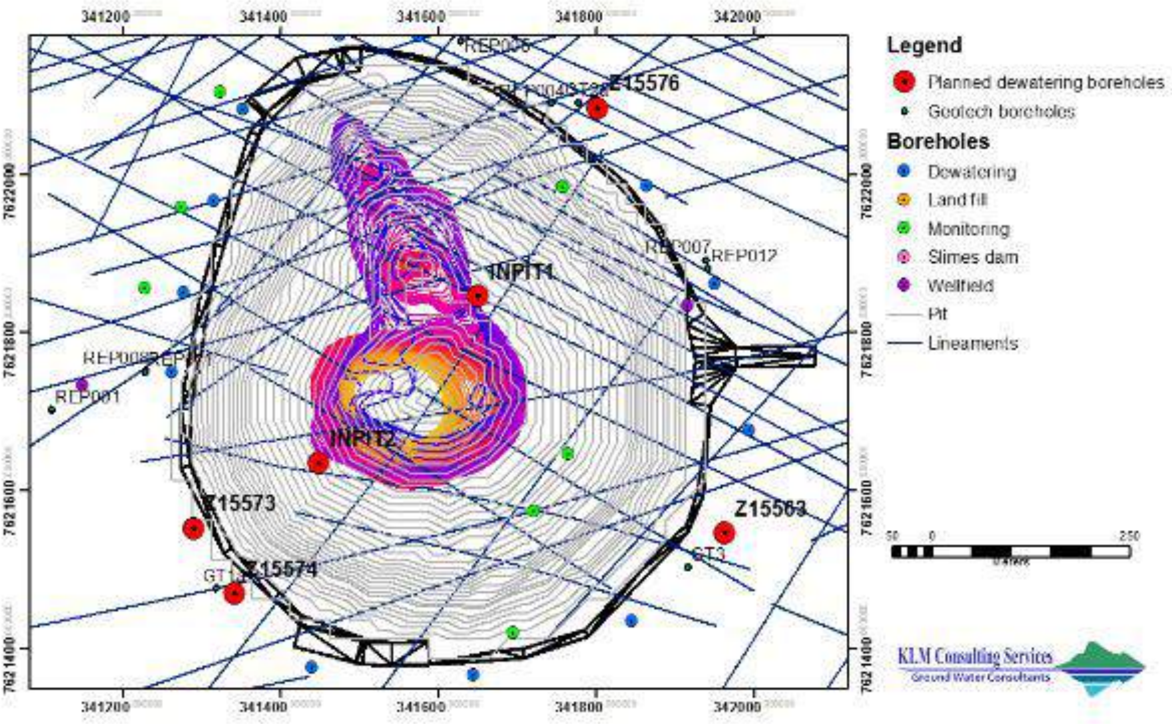


Figure 16.19 - Planned locations of four new perimeter boreholes and two in pit boreholes

The 16 boreholes installed in 2010 were constructed of slotted steel casing to reduce the mine start-up cost, their planned life was five years. The boreholes have also experienced encrustation and now have a very limited effectiveness. The planned dewatering volume from 16 boreholes was 325 m^3/hr (2.8 Mm^3/yr). The totals pumped in August 2017 were 240 m^3/hr (2.1 Mm^3/yr). Failure of the monitoring network (Vega probes linked to the plant SCADA system) has meant that the monitoring of dewatering effectiveness has not been used in the mine water management strategy, however the monitoring network is being upgraded.

Boreholes drilled to below the planned pit bottom and constructed with louvered, 2% copper mild steel casing will be added to the existing pit perimeter boreholes and be drilled to 410 mbs. This will increase efficiency and enable the Mosolotsane and structures below to be dewatered to pit bottom (c350 mbs). The use of additional in pit boreholes targeting the kimberlite contact will increase the effectiveness of dewatering. In discussion with mine planning there are two areas where in pit boreholes can be installed without interfering with production. Accurate monitoring of the pore pressures in the Mosolotsane and Tlhabala is essential to determine the effectiveness of the pumping on specific vulnerable units.

16.2.6 Proposed work for PFS

A two programme approach is proposed for the PFS:

1. The long term de-watering project (including the open pit, and in parallel with the surface and open pit dewatering), and
2. Studies and data collection (drilling, DTH geophysics, logging, packers, etc.) for the underground mine design, and characterization, models, etc.)

1. Tasks required to achieve a dewatered open pit and thereby enable smooth transition to underground mine are:

- Detailed planning of pit dewatering to 2020.
- Upgrade of water level and pumping monitoring to ensure effectiveness of dewatering.
- Upgrade of storm water controls and sealing of leakages to the mine from Waste rock dumps and ponds.
- Change of water balance strategy to ensure dewatering water, recycled water and wellfields are managed to optimize pit dewatering.
- Addition of 12 new generation pumping boreholes to 410 mbs targeting the Mea and granite formation (number to be confirmed from numerical modelling simulations).
- Additional of 2 angled pumping boreholes within the pit to intercept the granite contact with kimberlite.
- Upgrade of sump pumping and civil works to remove water from pit and boreholes timeously.
- Use of drain holes from within the pit to depressurise mudstones.
- Liaison and input to water permit and EIA process.

2. Tasks required to design underground dewatering for the underground mine design:

The hydrogeology above the Mosolotsane base is known therefore testing will be required for the units below and the update of the structures (both water bearing and barriers) exposed in the pit for the new structural model. The work assumes the new structural model will be available in January 2018.

Tasks required are.

- Numerical model to provide estimated of number of boreholes and timing to dewater the open pit (in progress).
- Drilling and testing of four deep pumping and four deep monitoring boreholes to granite contact.

- Liaison with Geotechnical investigations and use of combined effort drilling, interrogation and equipping of three core holes to install nested point piezometers (included DTH geophysics and if necessary packer testing). Assume Geotechnical holes will be used for all the permeability and porosity testing of the formations.
- Upgrade of monitoring network, data capture, interpretation and dashboards.
- Modelling of mudstone response to dewatering and depressurisation.
- Numerical modelling of hydrogeology of underground mine and scenaria assessment.
- Design of implementation of dewatering including, resource planning, controls and installation of outcomes based monitoring.
- Installation of dewatering infrastructure, monitoring network.

The PFS will continue to build on the investigation work done in the PEA. The main items to be addressed are:

- The discretization of the hydrogeological and geotechnical units that will require additional dewatering and support.
- Update of the structural and country rock hydrogeological model to 740 m.
- Collection of hydraulic and hydrochemical values for the identified units to 740 m.
- Installation of the pore pressure monitoring piezometers for identified units in the Tlapana, Mea, granite and kimberlite contact.
- Accurate dewatering and monitoring of the open pit.
- Upgrade of the monitoring network, data capture interpretation and presentation.
- Accurate management and detailed planning of the water balance.
- Integration of the hydrogeology and geotechnical design with the mine planning and engineering designs.

The PFS will therefore address the detailed hydrogeology and update the estimate of the cost to accurately dewater the open pit and underground mine.

16.3 Open Pit Mining

Determining the extent of the economic shell of the open pit was an integral part of the study. Whittle contains a function which allows the introduction of an underground mining cost which compares open and underground mining methods and determines the most economical method for each block to be mined. A range of Underground mining costs were used in various Whittle runs to determine the effect of increasing the cost of mining underground to the economic depth of the open it.

16.3.1 Geological block model used in whittle

The three-dimensional block model (Karowe_Block Model Consolidation_Separate ORE-WASTE.csv) was received from MSC on 15th September 2017. This contained the following fields:

- Volume – Total volume contained within the block.
- Tonnes – Total tonnes contained within the block.
- Carats – Total carats contained within the block.
- Dollars – Total dollar value contained within the block.

All the blocks had the following dimensions:

- 25 m in the X direction.
- 25 m in the Y direction.
- 12 m in the Z direction.

The model contents are shown in Table 16.13.

Table 16.13 - Block Model Contents

Model	Tonnes	Carats	Dollars	cpht
September	72 360 059	10 759 726	7 051 428 996	14.87

A wireframe of the surface (Figure 16.20) of the current pit position was supplied (May2017 Surface.dwg) and the block model was cut to this surface.

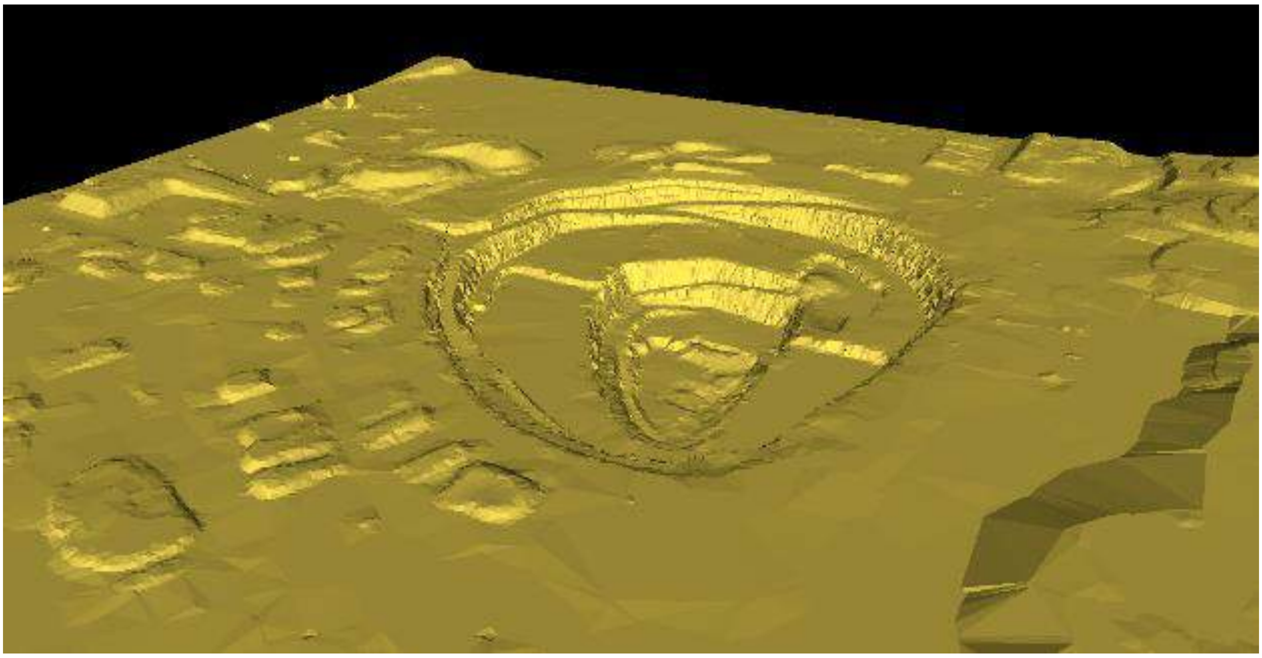


Figure 16.20 - Surface topography

16.3.2 Whittle runs

Input Parameters

The input parameters used in the Whittle runs are shown in Table 16.14 below. Unless specified these parameters were suppled to RHDHV by Lucara.

Table 16.14 – Whittle Input Parameters

Whittle Inputs	Unit	Value
Open Pit Mining Cost	US$ per tonne rock mined	3.26*
Mining recovery (% extraction)	Percentage	90%
Dilution	Percentage	5%
Cost Adjustment for Depth	US$ per tonne per meter	0.02
Mining bench height	m	12
Slope angles	Degrees	45º
Processing cost	US$ per tonne processed (RoM)	11.82
Overhead and marketing costs	US$ per year	2,920,000
Rehab Cost	US$ per tonne waste rock	0.01
Discount rate	Percentage/annum	8.26%
Revenue	Value set in Block Model	
Processing recovery	Percentage	92%
Mining Cost for Underground	US$ per tonne Kimberlite mined (RoM)	25**

** See Calculation of Open Pit Mining Cost*
*** See Variability of Underground Mining Cost*

Calculation of Open Pit Mining cost

RHDHV was given a spreadsheets of actual open pit mining cost to date (Mining Cost Yr to Date.xlsx) and tonnage mined (20170821 Karowe Open Pit Depletions_Jan-Jul.xlsx). From these an average mining cost per tonne was calculated as shown in Table 16.15.

Table 16.15 – Open Pit Mining cost Calculation

Description	Value
Total Kimberlite Mined Year to Date*	6 142 770
Total Mining Cost YTD (Pula)	207 265 988
Mining Cost Pula per tonne	33.74
Exchange Rate**	0.097
Mining Cost US$/t	**3.26**

* *Year to Date = Jun Qtr 2 2017*
** *Exchange Rate from XE.com 07-08-2017 PO.0967102*

A cost adjustment of US$0.02 per tonne per metre for mining at depth was included based on industry standards as well as a rehabilitation cost of US$0.01 per tonne waste mined.

Variability of Underground mining cost

In order to determine the effect of applying an underground mining cost to the open pit depth, a number of Whittle runs were completed at various underground mining costs. The resulting pit shells were then compared to the current Cut 2 mine design.

List of underground mining costs used:

- No underground cost (open pit only).
- US$25.
- US$36.82.
- US$40.
- US$45.
- US$50.
- US$55.
- US$60.
- US$100.

Initial estimation of underground mining methods for the proposed method was US$25/t. In order to compare an underground mining potential to the open pit mining potential, Whittle uses the underground mining cost as an optional processing stream. The actual processing cost of US$11.82/t is thus added to the US$25/t mining cost. Each block is then considered to either be mined and processed in the open pit, or mined and processed in the underground method.

Whittle run Results

The results of the various whittle runs are shown in Table 16.16.

Table 16.16 – Whittle Results

	UG Mining Cost US$/tonne	Rock Tonnes	Kimberlite Tonnes	Waste Tonnes	Strip Ratio	Value US$ / tonne
Option 1	**0**	352 283 594	41 343 739	310 939 855	7.11	94.71
Option 2	**25**	16 162 103	10 293 982	5 868 121	0.57	78.63
Option 3	**36.82**	47 966 750	19 669 509	28 297 241	1.44	84.27
Option 4	**40**	52 400 742	20 492 707	31 908 035	1.56	86.43
Option 5	**45**	60 111 417	21 732 941	38 378 476	1.77	87.71
Option 6	**50**	77 327 265	23 951 432	53 375 833	2.23	88.29
Option 7	**55**	92 769 662	25 625 852	67 143 810	2.62	89.45
Option 8	**60**	113 167 746	27 613 769	85 553 977	3.10	91.04
Option 9	**100**	317 744 636	37 819 887	279 924 749	6.06	94.48

Initial estimates showed an underground cost of US$25/t would be reasonable; in the Whittle runs this equates to Option 3 (including other operating costs). Since this was close to the existing Cut 2 design that was used in the current Life of mine, the PEA used this option going forward.

Figure 16.21 shows a graphical representation of the resulting pit shells (not all options are shown as some overlap).

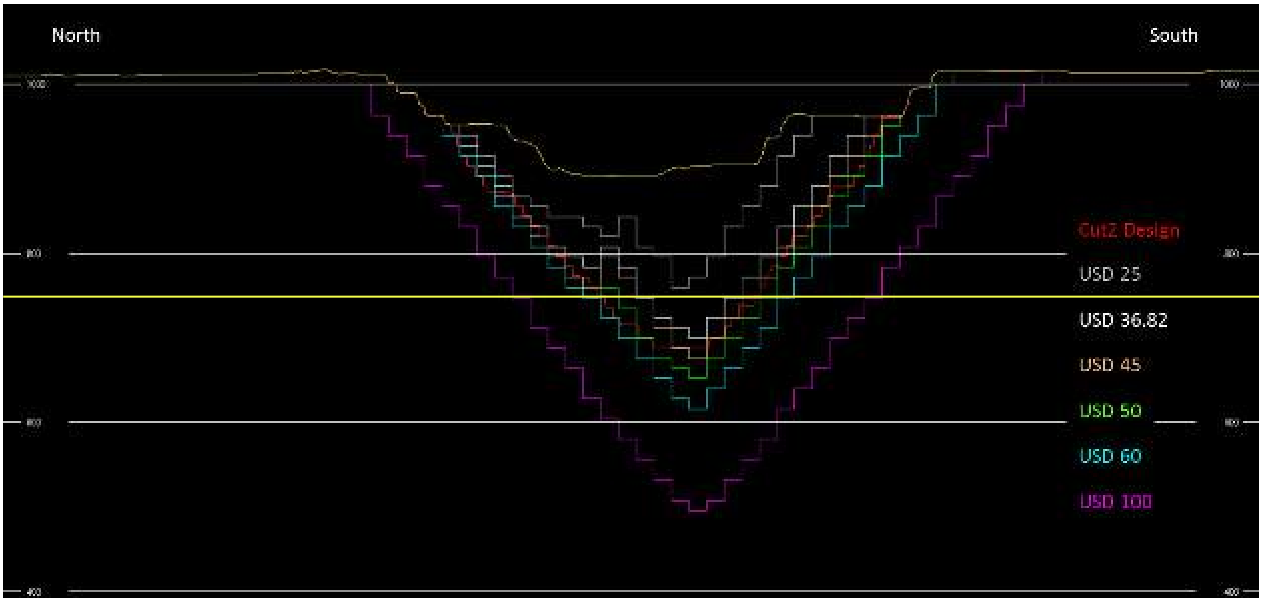


Figure 16.21 – Whittle Pits for varying underground costs

16.3.3 Observations from the whittle runs

The Whittle run results show it is reasonable to use the existing Cut 2 design and the existing Life of Mine Schedule for the PEA study. Figure 16.22 shows Cut 2 and surface topography as supplied.

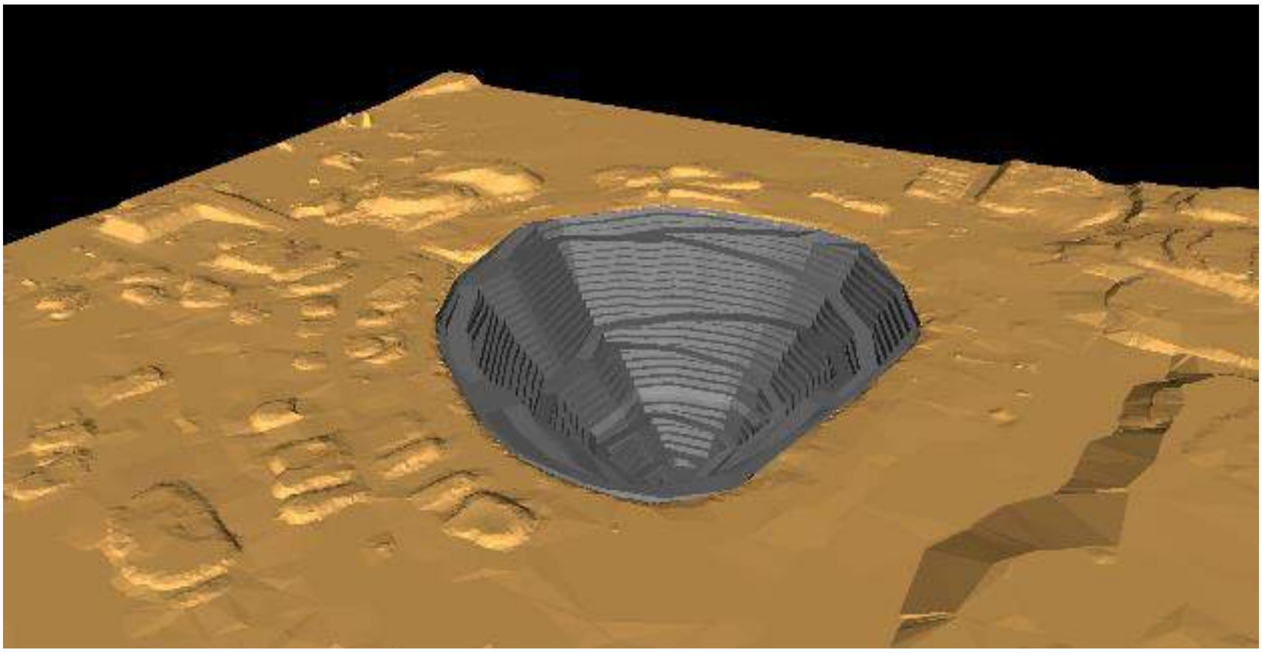


Figure 16.22 - Cut 2 Pit design with topography (looking South East)

However, observations were made during the process as follows:

Changes to the Block model

During this study revisions were made to the block model and it was noted that changes to the grade and tonnage of the kimberlite has an effect on the size of the resulting Whittle pit shells. Once the current drilling is analysed and included in a new block model, the final economic open pit depth will need to be re-determined during the PFS study.

Underground cost

As the underground cost is more accurately estimated, it is updated in the Whittle runs (somewhat of an iterative process). For example, if the total underground cost to be used in Whittle is US$60/t, the resulting open pit shell contains an additional 5.6 Mt of kimberlite and 29 Mt of waste to that contained within Cut 2. The average grade and value for this additional kimberlite has a grade of 13.23 cpht and an average value of US$95.07/t. This suggests that there may be a significant increase in waste stripping if the pit is to be deepened.

The practicality of deepening the pit must be considered. Stripping for Cut 2 is already underway and increasing the stripping to achieve another cut may actually have a negative effect on the mine economics due to an increase in waste mining.

16.4 Underground Mining

16.4.1 Background

Currently the Karowe AK6 Diamond pipe deposit is being mined by open pit and a Whittle run carried out by RHDHV has confirmed that the economic cut-off in comparison to underground mining is at 690 mamsl as determined previously by the mine. The Whittle run employed the understanding of the current mine costs for open pit mining including completing the Cut 2 push back and the likely direct costs for underground mining US$45/t mined.

The bulk mining method suited to the underground mining of the pipe, measuring an average 200 m diameter to 420 mamsl (590 mbs), for this PEA was either sub level caving (SLC) or block caving (BC). A high level cost comparison was carried out, encompassing the known geological, hydrogeological, geotechnical and the physical attributes of the deposit, indicated opting for SLC at the PEA stage. The operating cost advantage for BC was overridden by the perceived risk factors, discussed below, and high upfront capital of BC. The difference between the 2 methods will require an in-depth trade off exercise at the PFS (preliminary feasibility study) stage when improved data will become available in the geological, hydrogeological and geotechnical fields.

It is understood that the open pit will be able to maintain the current production rate of 2.5 mtpa until 2026. The principal objective for the underground operation will be to ensure economic maintenance of this production rate from 2026 onwards.

16.4.2 Mining method selection

The bulk underground mining method selected to mine the diamond kimberlite pipe measuring some 200 m in diameter is SLC. Initially the pipe will be partially offset from the pit bottom as shown in Figure 16.23, resulting in the top four sublevels being slightly offset from the pit bottom. These top levels plus two levels below the pit bottom will be mined by a derivation of sub level open stoping (SLOS) daylighting into the pit bottom.

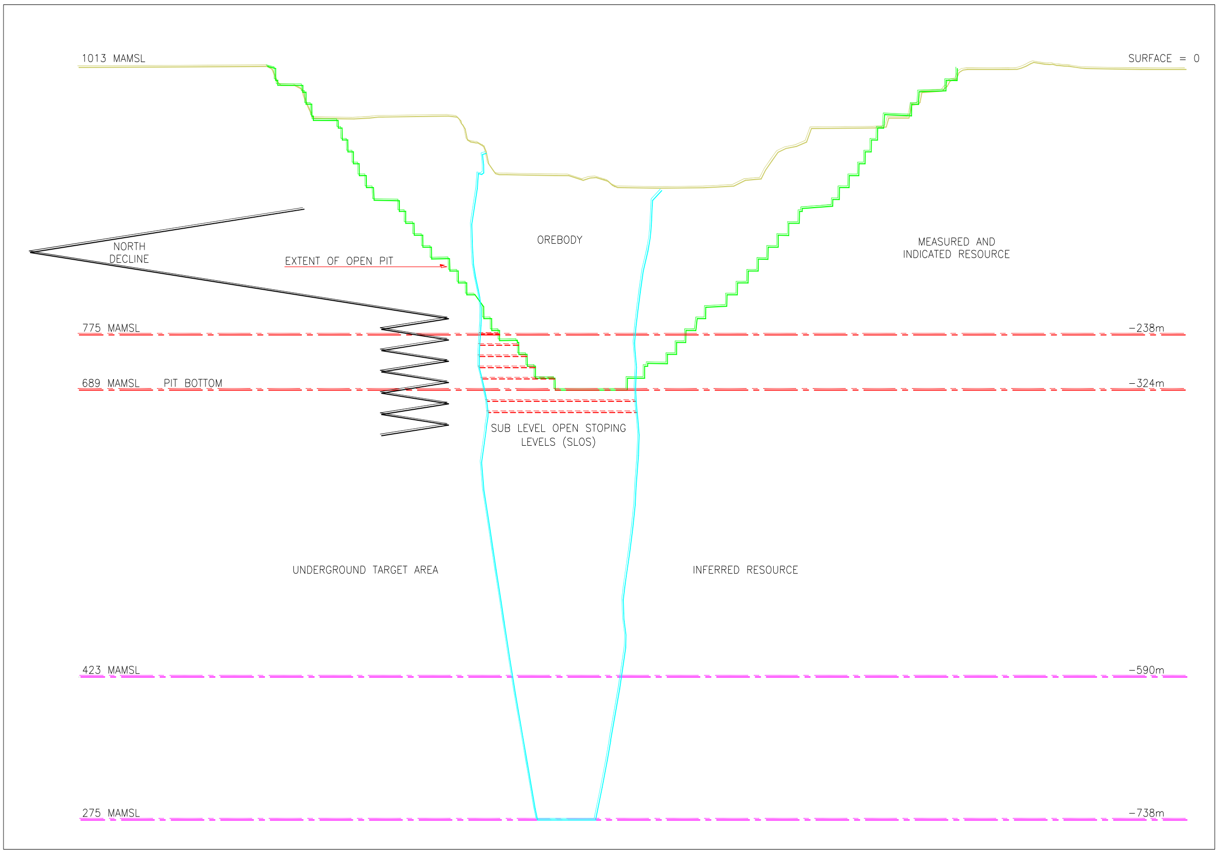


Figure 16.23 – Schematic section of the Sub-Level open Stope (SLOS) Top Levels

The following key factors were taken into account in the selection of the SLC method at this PEA stage:

- The uncertainty surrounding the extent of the hydrogeological conditions at Karowe. It is accepted that the water table has to be lowered to below the mining horizon before the access infrastructure and production can take place.
- SLC is a delayed capital option with the infrastructure installed progressively as the mining front proceeds downwards. The BC option requires high capital upfront for the total infrastructure.
- For the PEA, mining will be limited to a depth of 420 mamsl (590 mbs) which falls within the economic trucking range from surface obviating the need for a high capital upfront choices such as shaft hoisting or a decline conveyor system.
- The kimberlite rock strength at 156 Mpa is high with considerable homogeneity and limited jointing which is likely to inhibit caving. The competent ground conditions are positive for SLC mining but potentially

problematical for BC. Further investigation may lead to the necessity for pre-conditioning the kimberlite with hydro fracturing for BC mining. This potentially has both cost and time implications and it may be difficult to achieve the required production rate in time using BC.

- The quality and strength of the host rock is poor and may well be further compromised by water ingress, this could pose difficulties for the declines associated with the SLC option.
- Given the known kimberlite resource, overall the SLC option represents a lower extraction risk level than BC.

An in-depth trade off study comparing SLC to BC will be undertaken at the PFS stage.

16.4.3 Mine design

The mine design for both the upper levels SLOS and the following SLC mining will be similar once general caving of the host rocks commences. The sublevel interval will be 25 m, which is the international norm for the method. Owing to the hardness and strength of the kimberlite at 156 Mpa it is intended to evaluate increasing the sub level interval to 30m at the PFS stage. This will be subject to maintaining the blast hole ring drilling and charging capabilities. The number of sub levels would be reduced by 9% and the blast hole ring shape would be improved to a more vertically centred ellipsoid that could benefit the draw characteristics.

The assumed competence of the kimberlite also allows the size of the extraction crosscuts to be developed with dimensions of 5.0 m wide x 5.0 m high enabling use of the large Sandvik LH621 (21 tonne LHD) loader. The spacing of the crosscuts skin to skin are set at 8.0 m to optimise the draw ellipsoids as shown in Figure 16.24. The flexibility of the method allows adjustments to be made to the sub level intervals and crosscut spacings to suit localised geotechnical conditions. This will apply particularly to the upper level SLOS access levels.

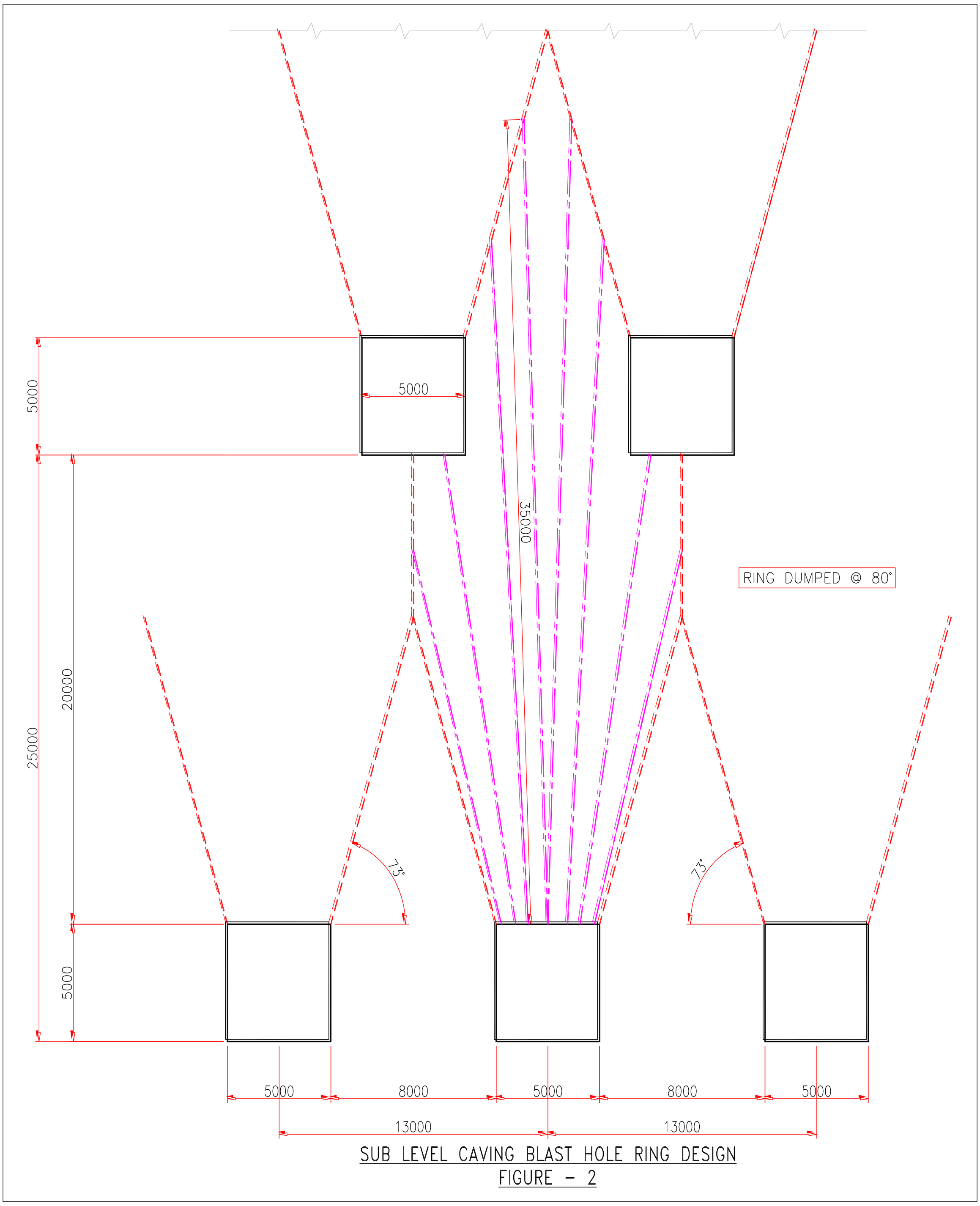


Figure 16.24 – Sub-Level Cave (SLC) Blast Hole Ring Design

Sub Level Open Stoping (SLOS)

The upper levels will be designed to extract the kimberlite pipe lying to the west side and immediately below the bottom of the pit. Conditions in the pit walls will dictate the final details of the tunnel positions to locate the blast hole ring positions for the pipe extraction. To provide a safety barrier from sloughing from the pit side walls and the extraction tunnels, a delayed draw procedure will be adopted to provide a kimberlite cushion between the pit and the tunnels. This involves only drawing the blast swell from the initial rings and levels to provide a kimberlite safety cushion or blanket.

The cushion can be extracted from the bottom level. Experience indicates that should a disciplined tonnage/grade draw control procedure be maintained a good kimberlite extraction at low dilution levels can be achieved.

Sub Level Caving (SLC)

As with the SLOS method the assumed competence of the kimberlite also allows the size of the extraction crosscuts to be developed with dimensions of 5.0 m wide x 5.0 m high and the spacing of the crosscuts skin to skin at 8.0 m to optimise the draw ellipsoids, Figure 16.24. The flexibility of the method allows adjustments to be made to the sub level intervals and crosscut spacings to suit localised geotechnical conditions. This will apply particularly to the upper level initial SLOS access levels.

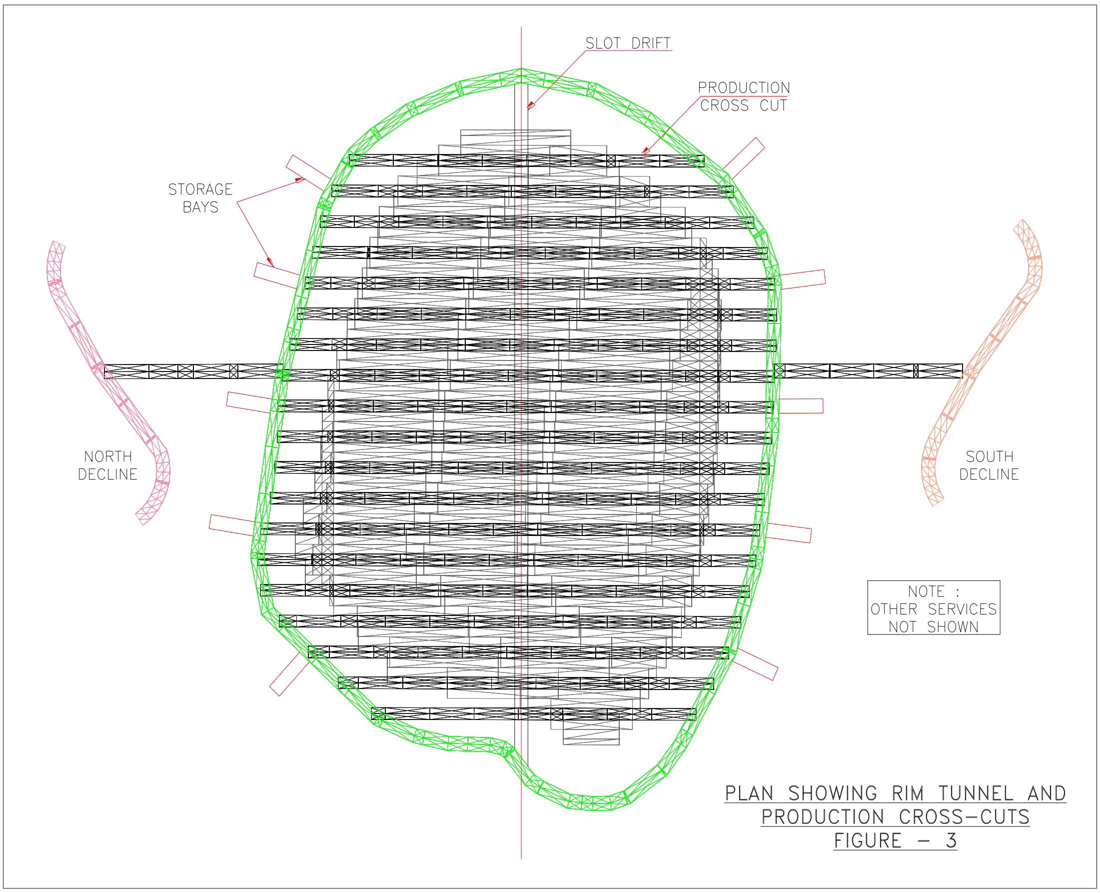


Figure 16.25 – Schematic plan of a Sub-Level showing the rim tunnel and production cross cut

The SLC blast hole rings are currently planned for a common 8 hole ring with 102 mm holes at a 2.3 m burden and 2.0 m hole spacing. At PFS level further evaluation of the ring design will be carried out to optimise both the fragmentation and to limit blast damage to the diamonds. Data from the open pit bench blasting will be analysed for direction.

To maximise the number of operational crosscuts on a sub level, the slot for the pipe will be placed in the centre to double the number of working draw points to 38 as shown in Figure 16.25. The slot bifurcates the pipe in an east/west direction with the crosscut draw points cleaned back to the access ramps at the north and south ends. The crosscuts are relatively short in length commencing at 100 m at the slot position.

The sub level extraction crosscuts are accessed from a rim tunnel with dimensions of 6.0 m width x 6.3 m height (shown in green in Figure 16.25) developed around the pipe connecting to the 2 surface access ramps and ventilation raises.

It is proposed to continue using the delayed draw procedure on the lower levels where the extraction percentage can be increased to 100% while maintaining the initial kimberlite cushion provided by the limited extraction from the upper levels. This process first developed at the Granduc Mine in British Columbia, Canada effectively removes the main cause of dilution with standard SLC where the ring kimberlite columns are drawn against a cushion of waste.

For this PEA it is estimated the dilution can be reduced from the standard 35% to some 12% if the delayed draw process is adopted. The kimberlite loss is maintained at 15% due to the loader constraints in the draw points as well as the losses from the dynamics of draw of angular rock under pressure.

16.4.4 Underground primary access

In this preliminary design, the ramps are developed from surface boxcuts and traverse poor ground conditions over some 2000 m each to reach the first underground mining levels. It is proposed that box cuts and ramps be started at an early stage from surface. This will also assist the dewatering programme to lower the water table. An alternative access by means of establishing portals in the pit will be investigated at the PFS level of study. It will be necessary to ensure that the ramp development does not negatively impact pit operations. The current pit shell is shaped to mine all 3 pipe lobes, the South, Central and North lobes. As currently planned the underground mining will be restricted to the much larger South Lobe which opens the opportunity to establish the ramp portals in the North and Central Lobe shell area and this should be incorporated into the planning for the Cut 2 push back. This evaluation should take into account the timing for this option to ensure seamless continuity of production from pit to underground production.

Owing to the weak ground or water bearing conditions in the bulk of the host rocks it is proposed to adopt an innovative access and rock handling method for the underground SLC operation. The kimberlite from the crosscut draw points will be directly loaded into trucks for hauling to the existing surface stockpiles or crushing facilities. It is not intended to use ore passes at any stage in the rock handling system. SLC is a top down method with the infrastructure developed progressively in tandem with the production levels. This option provides for a low upfront capital requirement compared to the shaft option requiring a high upfront capital spend.

It is planned to develop twin 6.0 m wide x 6.3 m high ramps to cater for large haul trucks such as the Sandvik TH663, 60 tonne unit loaded by LH621, 21 tonne LHD's. The twin ramp system will be required to cater for the production rate of some 7,500 tonnes per day of kimberlite and waste.

The other main factors impacting the primary access decision include the requirement to lower the water table to below the depth of underground infrastructure, and the uncertain conditions of the host rock formations.

16.4.5 Secondary access

The main feature of the secondary access are the rim tunnels circumventing the kimberlite pipe to connect the SLC crosscuts to the surface access ramps and services on each sub level, Figure 16.26. The tunnel is to be mined 6 m wide x 6.3m high for loading the 60 tonne trucks transporting the kimberlite and waste to surface and provide services to the cross cuts. To facilitate loading, the 21 tonne LHD's will be fitted with ejector buckets for end loading anywhere along the rim tunnel. The 2 access ramps will be designated for one way traffic obviating the need for passing other than slow moving service vehicles travelling in the same direction. Passing bays will be positioned at 85m centres.

Depending on the ground conditions in the host rock or in the pipe perimeter the rim tunnel can be placed to suit local rock conditions. This can be decided when the ramps reach the sub level elevations. If it is considered preferable to place the tunnel in the kimberlite pipe, the resultant ring layouts will have a limited negative effect on kimberlite available for mining.

16.4.6 Mine planning

The mine design was done in Datamine Studio 5D Planner. At this level of study, only the main infrastructure was designed. Table 16.17 lists the infrastructure that was designed and Figure 16.26 shows the isometric view of underground design.

Table 16.17 – Underground Infrastructure designed in Studio 5D Planner

Development End	Excavation Size	Rate of Mining
North Ramp	6m x 6.3m	Varies on Ground Condition
South Ramp	6m x 6.3m	Varies on Ground Condition
Access Crosscut	6m x 6.3m	60m/mo
Rim Tunnel	6m x 6.3m	60m/mo
Production Crosscut	5m x 5m	60m/mo

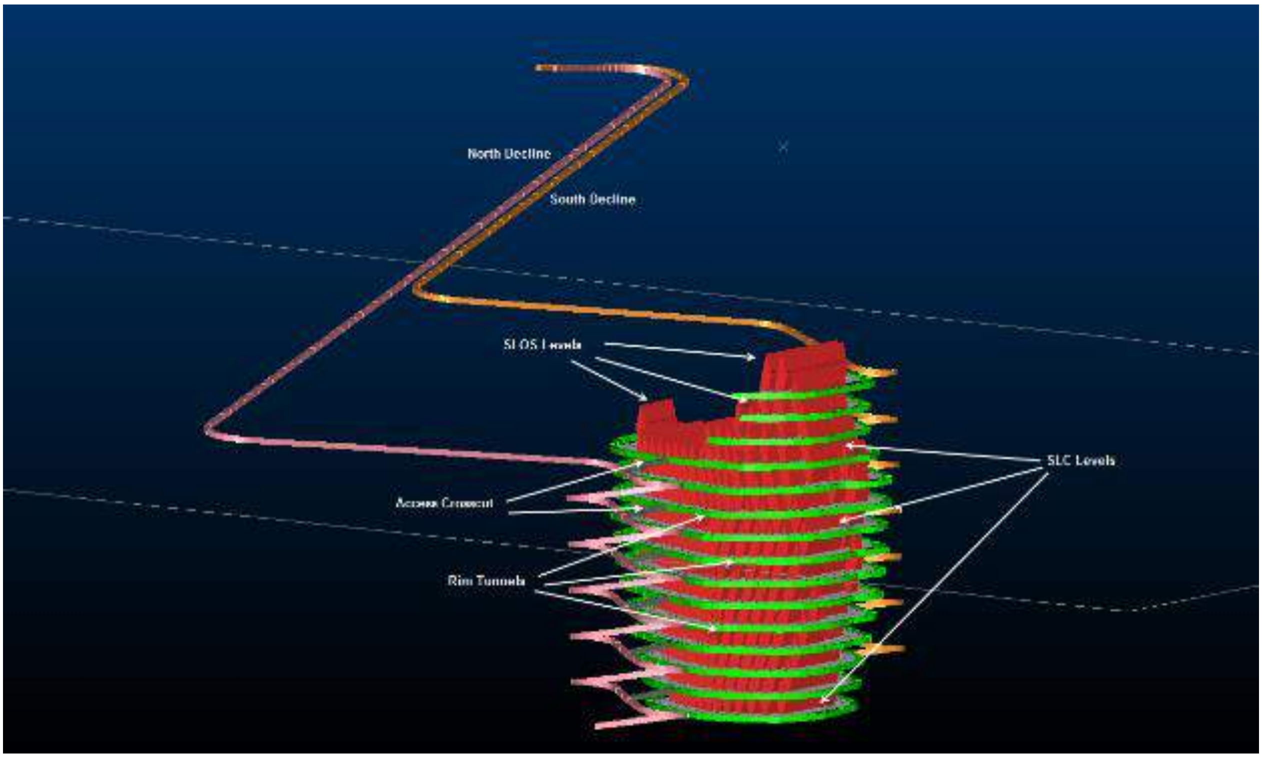


Figure 16.26 - Isometric View of Underground Design

A 15% waste development factor was added to the development to account for waste that was not designed into the layout and accounts for remuck bays, passing bays and service bays underground.

The main development ends were sequenced in 5D Planner and this information was sent to the EPS Scheduler to use for the Mine schedule. The design was also evaluated against the Geological Block model to determine the grade, tonnes and dollar value per tonne.

16.4.7 EPS schedule

The sequenced and evaluated data was sent to EPS (Enhanced Production Scheduler). EPS outputs the following information in a time based schedule:

- RoM Tonnes.
- RoM Carats per tonne.
- RoM dollars per tonne.

The following calculations were done in EPS:

- RoM Tonnes equal in-situ kimberlite tonnes plus 12% of waste dilution at zero grade.
- A kimberlite loss of 15% was deducted from the in-situ carats per tonne to determine the RoM carats per tonne.
- Dollars per tonne were reduced by 15% to determine the RoM dollars per tonne.

The production build up shown in Table 16.18 was determined in EPS. The production build up is based on the SLC and SLOS being mined at 7,000 t/d. The major constraint in reaching the steady state annual mining target of 2.5 million tonnes is the time required to develop access to the kimberlite. The twin ramp system development rate varies according to the ground conditions. The first kimberlite is mined from the SLOS stopes, 3 years after the commencement of the portal and a steady state of mining is achieved 3 years thereafter (Table 16.18). Steady state is maintained for 9 years and begins to tail off after year 13. The design is scheduled in such a way that there are always 3 levels being mined ahead of the current production level. This is to facilitate the timeous dewatering of the mine and to ensure that steady state is maintained. Table 16.18 shows the underground production schedule.

Table 16.18 – Underground Production Schedule

Name	Totals	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
South Decline Metres	***4 902***	827	827	611	414	414	414	317	207	207	207	207	207	44	-	-	-	-
North Decline Metres	***4 841***	485	827	782	414	414	414	415	414	310	207	161	-	-	-	-	-	-
Access Crosscut Metres	***2 741***	-	-	176	179	386	381	284	299	313	224	229	133	138	-	-	-	-
Rim Tunnel Metres	***12 391***	-	-	481	887	1 805	2 022	1 866	1 066	911	876	848	825	803	-	-	-	-
Crosscut Metres	***37 965***	-	-	358	1 427	5 151	7 243	7 068	3 394	3 110	3 129	2 640	2 411	2 036	-	-	-	-
RoM Tonnes	***24 726 833***	-	-	37 357	457 104	1 242 109	2 487 983	2 487 589	2 471 729	2 455 653	2 454 909	2 427 720	2 488 695	2 460 606	2 272 565	960 535	22 281	37 357
RoM cpht		-	-	7.67	8.00	8.85	11.22	12.37	13.24	12.26	11.35	12.91	13.13	12.44	12.44	12.41	11.59	7.67

16.4.8 Draw control strategy and procedure

The SLC mining method requires the implementation and ongoing maintenance of a disciplined draw control procedures. This is to ensure that the draw control department and operating management continually know the state of draw with respect to the location of the kimberlite/waste interface. At steady state, the SLC method is based on drawing the blasted kimberlite rings against a cushion of waste which is usually the main cause of dilution in addition to waste entering from vertically above. Strict grade control on the draw can control the level of dilution for an acceptable kimberlite recovery. If properly implemented RHDHV believe that this mine can achieve an overall kimberlite recovery of ± 85% at a dilution of ± 12%.

16.4.9 Ground water ingress mitigation

The ground water ingress from the sandstone host rocks is the major concern expected to negatively impact mining the Karowe pipe from underground. The key objective will be to lower the water table in the zone covering the kimberlite pipe and supporting infrastructure well ahead of mining. It will be important to make the time sufficient to effectively dry the surrounding beds prior to excavation and support. The three level gap between development and ultimately stoping affords a 20 month lead time.

16.4.10 Mechanised equipment fleet

For this PEA, RHDHV have costed a primary mechanised equipment fleet from Sandvik. Although the current costs will be based on standard diesel equipment it is intended at the PFS stage to include equipment automation and electrification where appropriate.

Rock Handling:

As noted in Section 16.3, following the intent to transport the kimberlite and waste by truck to surface and with the understanding that the kimberlite pipe competency will allow the use of large loaders in the kimberlite drawpoints, it was decided to opt for the largest available rock handling units.

The main rock handling fleet, at steady state to cater for 7000 tpd kimberlite and up to 1400 tpd waste, will include five 21 t LHD's and four 63 t trucks on the top sub level at 773 mamsl and up to eleven 63 t trucks on the 14th sub level located at 420 mamsl.

Drilling:

The development, rock bolting and long hole drilling drill rigs recommended by Sandvik include:

- Development Drill Rig; DD421E Twin Boom with RD525 drifters, multi voltage 380-1000v 160 kW motor.
- Long Hole Production Drill Rig: DL421 Single Boom with HL1560ST rock drill, 380 -690v 90 kW motor.
- Support Rock Drill Rig: DS421 cable bolter with HL510LH-38 (R32) rock drill with cement loading system, 380-690v 55 kW motor.

Support Equipment:

The main items in this category will include the Getman charging units, shotcrete placers with Agicar transporters, Caterpillar Telehandlers replacing slow scissors truck units and multi-purpose vehicles (MPV's) with a full range of cassettes.

Workshop and Service Bays:

The main workshop for the mechanised equipment will be on surface with use made of the open pit maintenance facilities where possible. The workshop will cater for the monthly services and major repairs. Underground service bays will be provided for daily services to drilling rigs and support equipment as required off of the sub level rim tunnels. Use will also be made of mobile rig servicing, refuelling and oils dispensing. With the mining front moving continually downwards emphasis will be placed on providing mobile facilities for frequent relocation of the service bays to lower levels.

16.5 Ventilation

16.5.1 Introduction

Planning for mining at depth was subjected to calculations, and eventually to a Ventilation of Underground Mine Atmospheres (VUMA) model to confirm the accuracy of the results.

Data available indicates that a large number of mobile equipment, especially 63 tonne trucks with 567 kW engines each, will be required to transport the broken kimberlite from the underground levels to surface.

To ventilate these mobile units, an internationally acceptable air factor of 0.11 m^3/s per rated machine kW is used. With the disregarding of small utility vehicles and drill rigs, the total air requirement was calculated to be 880 m^3/s, based on 7,997 kW or equipment.

Air velocities used in line with international accepted standards show that the two declines from surface will be able to carry 567 m^3/s, meaning the maximum depth that can be mined with the allocated equipment is down to 365 mbs (648 mamsl). However, the indicated depth of mining was however 590 mbs (420 mamsl), leaving a shortfall of 304 m^3/s.

Return air from the production areas will be forced into the open pit.

16.5.2 Manual calculations for mobile equipment and required ventilation

The data was subjected to manual calculations, and the heat load for the mobile equipment was established as shown in Table 16.19 below. This table also indicates the air requirement per level.

Table 16.19 – Mobile Equipment Heat Load

Depth		Equipment required	Equipment kW		Air / kW	Air required
240	mbc	5 LHD, 4 trucks	4028	kW used total	0.11	443.1
265	mbc		4028	kW used total	0.11	443.1
290	mbc	5 LHD, 5 trucks	4595	kW used total	0.11	505.5
315	mbc		4595	kW used total	0.11	505.5
340	mbc	5 LHD, 6 trucks	5162	kW used total	0.11	567.8
365	mbc		5162	kW used total	0.11	567.8
390	mbc	5 LHD, 7 trucks	5729	kW used total	0.11	630.2
415	mbc		5729	kW used total	0.11	630.2
440	mbc	5 LHD, 8 trucks	6296	kW used total	0.11	692.6
465	mbc		6296	kW used total	0.11	692.6
490	mbc	5 LHD, 9 trucks	6863	kW used total	0.11	754.9
515	mbc		6863	kW used total	0.11	754.9
540	mbc	5 LHD, 10 trucks	7430	kW used total	0.11	817.3
565	mbc		7430	kW used total	0.11	817.3
590	mbc	5 LHD, 11 trucks	7997	kW used total	0.11	879.7
615	mbc		7997	kW used total	0.11	879.7
Total air required to ventilate trackless equipment (rounded, m^3/s)						**880.0**

mbc = mean below collar

As mentioned above, the declines can only carry 567 m^3/s at full capacity.

Additional air must then be supplied to the workings via raisebore holes from surface. To supply this amount of additional air will require two 3.6 m diameter holes to the deepest workings. Alternatively, the declines must be enlarged by 19 m^2 each.

Table 16.20 below shows the detail as well as design criteria.

Table 16.20 - Air supply detail and design criteria

Criteria	Detail	Unit
South decline to 240 level	1684	m
North decline to 315 level	2380	m
Number of declines	2	
Excavated size of each decline	36.0	m^2
Designed downcast velocity	6 ~ 8	m/s
Preferred downcast velocity	6.00	m/s
Total dc capacity @ 6 m/s	432.0	m^3/s
Total downcast capacity @ 8 m/s	**576.0**	**m^3/s**
Rim tunnel half level length	435.0	m
Rim tunnel excavation size 6.0 * 6.0	36	m^2
Air required per rated kW diesel	0.11	$m^3/s/kW$
There is a shortfall of air of	**304.0**	**m^3/s**
Number of downcast shafts required	2	
Each shaft to carry	152.0	m^3/s
Downcast shaft velocity (ideal)	15	m/s
Theoretical shaft area	10.1333	m^2
Recommended shaft diameter	**3.6**	**m**
Alternative enlarge the declines by	19	m^2
Drill it down to 590 m intersection to get air into bottom ring tunnel		

Since the open pit will be used as return airway from the underground workings, it will be required to force air into the mine. Naturally the air will tend to upcast from the pit through the ramps to surface due to the pressure difference and system resistance. The fans chosen need to handle the required airflow of up to 4 kPa. All the mobile equipment need to pass through this fan installation. It is impractical to install doors or even brattices in such a travelling air / roadway, and it was opted to install a jet fan to overcome the velocity pressure of the main fan installation to prevent recirculation. It is a well-known method and used in many mines.

16.5.3 Modelling

To test the hand calculations and to determine the thermal exposure limits in the bottom of the mine at full production, a simple VUMA model was constructed.

Challenges experienced were inter alia to supply adequate air for a single end to ventilate an LHD with an engine capacity of 352 kW each. The air required is 39 m^3/s per end. There are two methods available to get that amount of air in a single tunnel:

- Install two 1,016 mm (40") columns from through ventilation to the face, using 75 kW fans to move the air.
- Install a single 1,400 mm (55") column in the development end, using 55 kW fans to move the air.

Both these options offer unique challenges. The two 1,016 mm columns will leave very limited space for especially the dump trucks to manoeuvre through the small opening, while the 1,400 mm column will be bulky to install. Since the mine will be on multiblast, all development ends need to be fitted with galvanised steel ventilation columns. Spiral tubing has much higher resistance, and is easily damaged.

For the sake of the model, it was decided to use the single 1,400 mm column and 55 kW fans.

Based on the scenario that, once in the reef plane, three ends can be ventilated in series from a single source of air, the development was modelled. At no stage the thermal stresses were found to exceed 30.0 ºC even with all eleven trucks and five LHD's operating.

Below are pictures of the modelling at various stages. In all cases, blue indicates intake or fresh air, and red contaminated air.

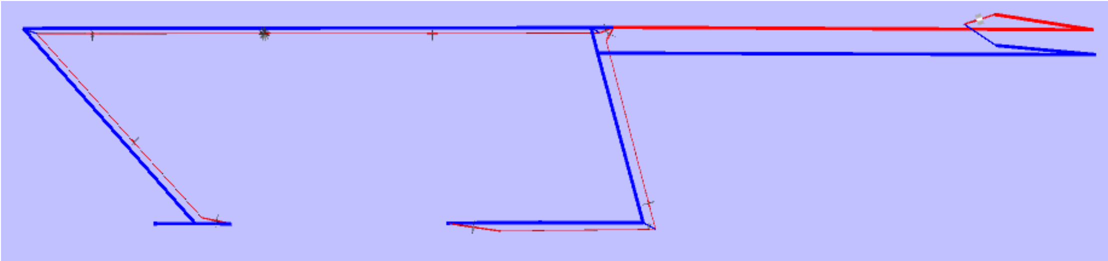


Figure 16.27 - Multiblasting the ramps to reef

The thin red lines next to the intake ramps are the ventilation columns and 55 kW auxiliary fans returning the air to ramp 1, with a suction fan situated on surface.

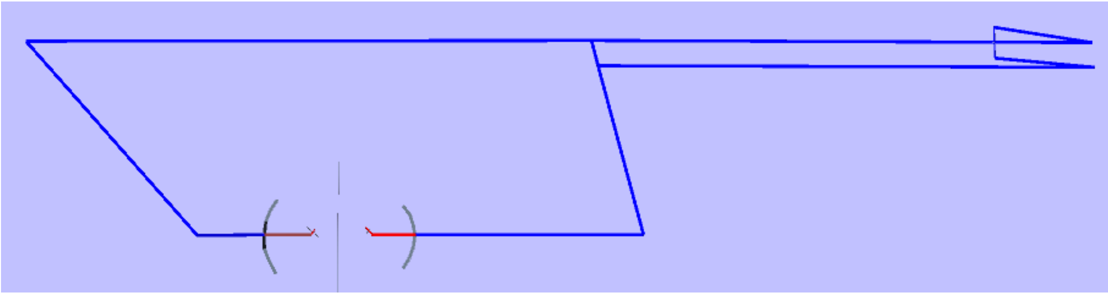


Figure 16.28 - Holing the ramp and on-reef development into the open pit

It must be noticed that the Northern (left) side holes on 315 m level, while the Southern) right side holes on 240 m level. This elevation difference was used all the way to the bottom during the modelling.

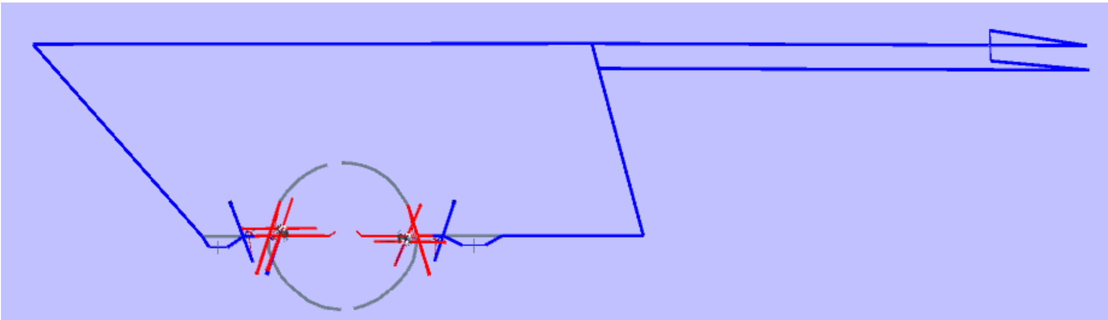


Figure 16.29 - Rim tunnel and on reef development started

The production tunnels were specified to be 5.0 by 5.0 m, making the supply of adequate ventilation to the face even more challenging.

Note the main force fans are installed near the entrances of the reef horizon, and will remain there for the life of the mine.

The ramps to the level below are also started to be developed.

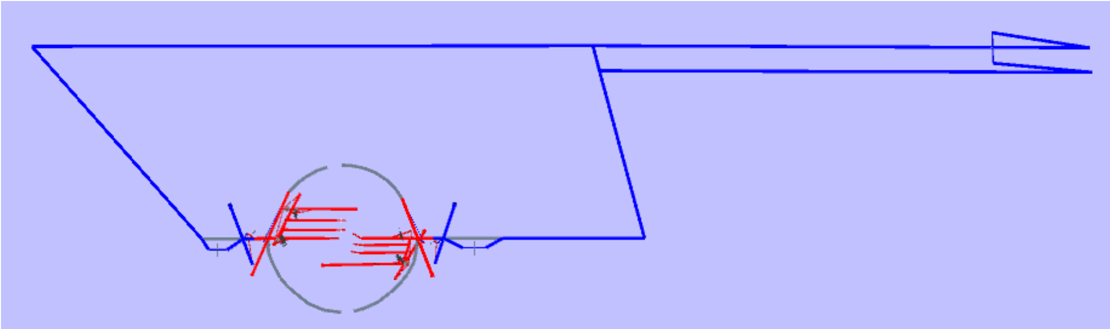


Figure 16.30 - Extension of the production tunnel development.

Three ends can be ventilated from a single source of air. Should it be practical to enlarge some areas to cater for ventilation column crossings, more ends can be ventilated simultaneously.

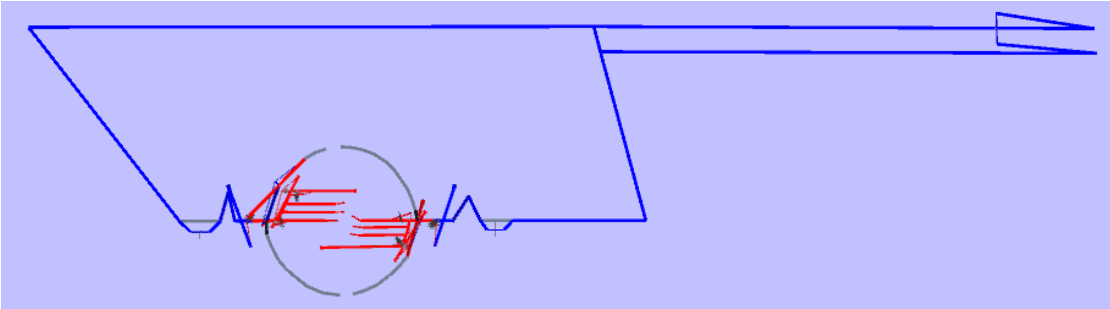


Figure 16.31 - Development on 440 and 365 levels.

The method of ventilation repeats itself to ensure that the development below is completed by the time the caving is completed above.

At this stage the temperatures are still within limits, but the downcast raisebore holes should be drilled to cater for additional intake air.

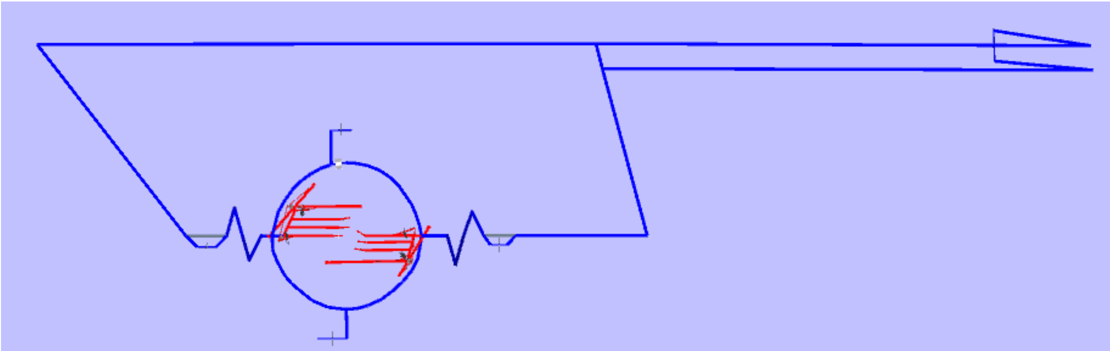


Figure 16.32 - Production on 590 level on both sides.

The downcast shafts on East and West sides are commissioned with main fans on surface, to supply additional air for the bottom section of the mine.

16.5.4 Considerations

- It will be advantageous to reconsider the development end sizes to safely accommodate ventilation columns and mobile equipment.
- Less air can be used if it could be possible to reduce the number and / or size of the mobile equipment, especially the trucks.

- The model indicated that the mine can be safely ventilated with only 550 m^3/s of air. The reason for this is that, once the ramps and horizontal development holed into the open pit, through ventilation is used for mining, and no dedicated return air ways are required.
- With less air required as shown by the model, the downcast raisebore holes should not be necessary. This, however, increases the underground fan pressures.

16.5.5 Conclusion

The mine can be safely ventilated to 590 mbs. Further optimisation of the ventilation requirements is envisaged at PFS levels of study.

17 Recovery Methods

17.1 Mineral Processing Plant

17.1.1 Flowsheet and process description

The Phase 2 upgrade is described in Section 13 and the resulting Process Diagram is shown in Figure 17.1.

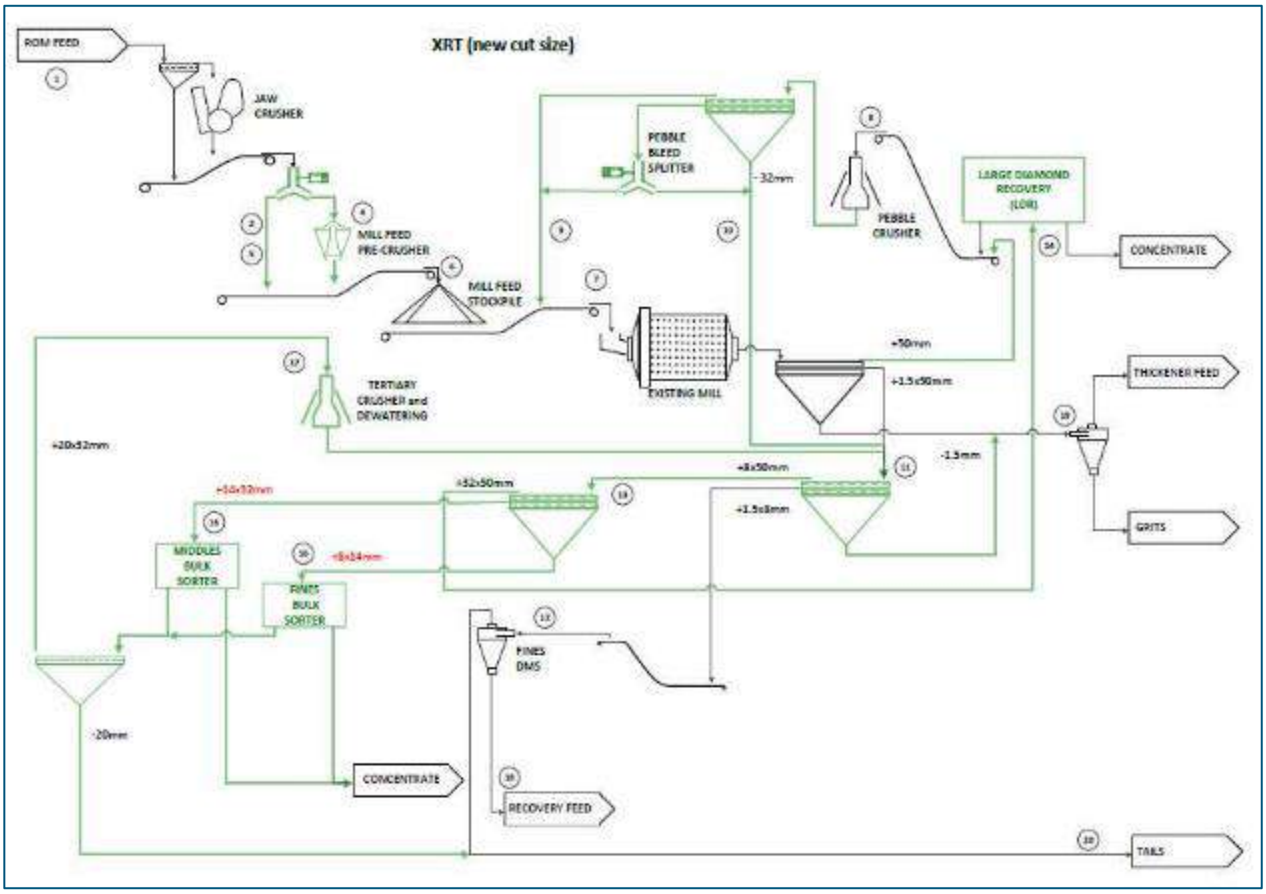


Figure 17.1 – Phase 2 Process Flowsheet

The Phase 2 upgrades were intended to:

- Protect and enhance recovery of large diamonds.
- Enhance comminution performance to maintain design throughput with harder kimberlites.
- Minimise recovery yields when treating these harder kimberlites.

Whilst these process enhancements should cater for the expected increase in kimberlite hardness, with the change from open pit to underground and increasing depth, hardness may increase further putting additional stress on the comminution circuit and recovery yields. Information on kimberlite characteristics that will affect plant throughput and recoveries must be collected and assessed during the ongoing drilling and sampling campaign and will give an indication whether or not further enhancements will be required to successfully treat potentially higher yields from underground kimberlite. Any additional comminution requirements would lead to an increase in energy requirements.

Phase 3 process upgrades have been completed and have been in operation since Q3 2017. These upgrades include an XRT circuit treating +50-125mm material, prior to milling, and enables the recovery of larger diamonds as early as possible in the process in addition to reducing the risk of diamond damage. A new XRT circuit has also been introduced to treat the 4-8mm fraction, previously sent to DMS, thus reducing the load on the DMS to cater for higher yield material expected in future.

Once all the process enhancements have been fully optimised, planned maintenance requirements for all the newly commissioned sections should be reviewed in order to minimise unplanned stoppages. Overall Plant Utilisation (OPU) will need to be maintained at target levels of 85% of design tonnage throughput, to ensure a suitable base for the treatment of the future underground resource.

Slimes and tailings disposal facilities have been designed to accommodate currently forecast open pit production. The underground project will potentially more than double the life-of-mine and sufficient surface area must be allocated to extend dam and dump footprints. This may require the purchase of additional land contiguous to the current surface rights.

17.1.2 Recommendations

As the underground project progresses to the next level of study, it is recommended that a Geometallurgical data base be established in order to record any pertinent data or information that is gathered from the exploration/drilling programmes. In addition, data from current operations should be recorded as the pit goes deeper and trends noted in kimberlite hardness, density and particle size distribution for example. This database would then be used to provide the expected underground ROM feed characteristics envelope as inputs for modelling the process. Thereafter plant performance can be predicted and enhancements to the flowsheet highlighted which may be required to achieve target throughputs and recoveries. The potential for tramp metal to report with the plant feed must also be assessed once the mining method has been chosen in order to cater for additional detection and removal equipment as necessary.

18 Project Infrastructure

Karowe Mine as an active operating mine is serviced by existing infrastructure. The following describes the main infrastructure currently in place on the mine:

The property is accessed by 15 km of well-maintained all-weather gravel road from the tarred A1 Letlhakane to Orapa road. International airlinks are available in Francistown and Gaborone. The mine also has its own private airstrip constructed of gravel. It is licenced for aircraft with a gross weight up to 5.7 tonnes.

Employees live in Letlhakane and are transported to the mine by bus or by private vehicles.

The open pit operation is serviced by haul roads within the mine boundaries.

The current mine infrastructure includes a metallurgical plant, administrative offices, mine vehicle workshops, slimes dam, and various stockpile and waste dumps. Existing contractor areas include the mining contractor, the contractors undertaking process operation and maintenance as well as process upgrades.

Water for the mine is provided by 16 wells situated around the periphery of the mine. These wells are part of a managed open pit dewatering strategy which is aligned to environmental and geotechnical requirements of the mine.

Karowe Diamond Mine generates DMS coarse processed kimberlite (tailings) which are disposed of on a dry dump by conveyor. Fine processed kimberlite (slimes) disposal is done into impoundment dams built and contained with mine overburden material.

Electricity is supplied to the mine by the national grid serviced by the Botswana Power Corporation (BPC) from a substation at Orapa.

A fuel depot is located on the mine. The depot is serviced by a local fuel provider.

A land line telecommunication link to the mine from Letlhakane is provided and serviced by the Botswana Telecommunications Corporation (BTC). In addition, there is cellular network currently operated by Orange Botswana accessible at the mine.

18.1 Surface Infrastructure

New surface infrastructure will be required over and above the existing infrastructure currently servicing the existing mine. The following describes the major new and/or upgraded infrastructure required:

18.1.1 Roads and air access

The existing roads and air access routes already servicing the mine will be sufficient to continue servicing the mine. No capital upgrades are foreseen.

18.1.2 Portals

The underground workings will be accessed by two declines. A single portal will be required on surface for each of the declines.

18.1.3 Workshops

A workshop on surface currently services the open pit mining fleet. While the existing workshop infrastructure can be reused when the underground mine starts production, additional workshop infrastructure will be required to support the underground development fleet as well as the production fleet. Fuel storage and distribution capacity will need to be increased during the underground development phase.

18.1.4 Water handling

The volume of water produced from the underground workings will increase the amount of water that needs to be treated on the mine. The existing volume of water from the boreholes provides sufficient water for the process plant and dust suppression. Additional water from underground will be discharged into the environment or placed into evaporation ponds. Water discharged into the environment will be treated in a plant to meet the legislative requirements.

Other water infrastructure will include dams and settlers to remove the suspended solids from the water pumped from underground. A revised water balance will be undertaken in the PFS phase of study.

18.1.5 Change house and crush

A new change house as well as a lamp room and crush facility for underground staff to change and shower will be required for the underground mine. The change house will be equipped with a laundry.

The lamp room and crush facility will be used to store and issue cap lamps and self-rescue units. The crush will be equipped with charging units.

18.1.6 Contractors area

A section on surface will be dedicated for the decline sinking and development crews. The area will be sized to receive and store services (piping, steel, cables etc) and equipment. A concrete batching plant will be required for the mine development. The area will include workshops for the underground fleet.

18.1.7 Bulk power supply

Botswana Power Corporation (BPC) supplies power to Karowe at 33 kV via a single overhead line from Orapa

A power supply agreement was signed between Boteti and BPC in September 2011. The agreement terminates on the 9th of October 2023. The agreement does make provision for Boteti to extend the contract for a term equal to the extension of the mining license.

The notified maximum demand (NMD) is 12 MVA.

Two 15MVA 33/11kV stepdown transformers are installed at Karowe. The primary reticulation on the mine site is at 11kV.

The maximum demand (MD) for the period May to June 2017 is approximately 7 MVA.

The estimated load of the underground operation up to 2023 is 12.2 MVA after power factor correction. This is slightly more than the current NMD and the increase will have to be communicated to BPC. The MD before correction is 13.6 MVA which is within the capability of the transformers.

After 2032 the MD increases to 13.8 MVA with power factor correction. This is still within the capability of the transformers, the MD before correction being 15.4 MVA, which exceeds the capability of the transformers. Allowance is made in the capital schedule to increase the transformer sizes to 20 MVA. This will have to be arranged with BPC.

18.2 Underground Infrastructure

18.2.1 Underground dewatering

Inflows into the underground operations will be from fissure water originating from the local aquifers, precipitation through the open pit, and service water. The water will be collected into dams located adjacent to the declines. The partially settled water will be staged pumped to surface. The water pumped to surface will be utilised in the plant, dust suppression and service water supply.

18.2.2 Vehicle service and refuelling bays

Underground service bays will be required to carry out daily inspections, washing, minor services and refuelling of equipment such as LHD`s and Drill-Rigs.

18.2.3 Power and communication

Power to the underground will be supplied from a new decline substation. The decline sub-station will be supplied from the existing main intake substation via two new circuit breakers in the substation. The PFC system will also be upgraded for the additional load.

Two feeders will be installed down the declines. An underground substation will be installed at the bottom of the decline. The underground substation will supply a ring feed down the declines to the mining levels. Mini-substations will be installed on the operating levels. For the pump stations, a dedicated sub-station will be installed at each pump station.

Underground communication will be via a leaky feeder system. A fibre-optic system will also be installed to provide communication and control between surface and each pump station PLC.

18.3 Tailings Storage Facility

18.3.1 Background

Karowe Diamond Mine generates DMS coarse tailings which are disposed of on a dry dump by conveyor, and fine tailings (slimes), which are disposed of onto 2nd Generation impoundment dams built and contained with mine overburden material. The existing fine tailings facility consists of four compartments, which are deposited and raised on a continuous cycle until it reaches the final elevation of 1041m above sea level.

A similar complex is planned to be constructed on the southern side of the current facility and operated on the same principles as the current facility. There is sufficient capacity until the year 2026. The following section is a brief overview of the future facility (post 2026) required for underground production, where the following is considered:

- Life assessment and size.
- Methodology.
- Infrastructure.
- Estimated cost.

18.3.2 Design Criteria

The following design criteria is applicable:

- Life required = 15 years
- ROM = 2 650 000 tonnes per year
- Recovery = 5%
- Tailings generated = 2 517 500 tonnes per year
- Coarse tailings:
  - Production = 94 791 tonnes per month
  - Assumed density = 1.7 kg/m^3
  - Volume = 55 759 m^3 per month
  - Volume for life = 10 036 765 m^3
  - Max height (1045) = 30 m
  - Slope angle = 1:3
  - Estimated area required = 48 ha
- Fine (wet) tailings:
  - Production = 115 000 tons per month
  - Assumed density = 1.35 kg/m^3
  - Volume = 85 186 m^3 per month
  - Volume for life = 15 333 333 m^3
  - Max height (1041) = 20 m
  - Impoundment wall slope angle:

- Inside = 1:2
- Crest width = 10 m
- Outside = 1:3

- Estimated area required = 121 ha

18.3.3 Layout

Due to boundary restrictions at the time of this study, the future complex will be placed west of the current and phase 2 facilities and south of the magazine area. From the information given, this area should be large enough for the expected 15 years of operations depending on the density and production rates. If additional space is required, the current and phase 2 top footprints areas are available for additional deposition space.

The coarse tailings will be placed east of the current facility and south of the current coarse dump. See Figure 18.1 for layout.

18.3.4 Methodology

The fine tailings will be placed behind a 20m high impoundment wall. The placement in the inside will occur by a spigot method system. A ring feed delivery line with a take -off every 30 m. This will assist with the pool control around the decant system. The facility is divided into 4 compartments. Each compartment consists of a blanket drainage system for seepage water and a self-decanting penstock tower for operational water. Free water decants through the penstock system where it will be pumped and reused in the plant. The penstock system consists of a top and side intake, which allows decanting of water, if the natural slope and shaping of the basin allows it. Alternatively, a floating barge with a sump system can be installed to decant water. With the low point created around the decant system, water back to the plant can be expected in approximately five days. Based on the assumed feed density of 1.30 t/m^3 an estimated 600 m^3 per hour will be placed where roughly 30% will be pumped back to the plant area. A deposition cycle will be followed in each compartment to allow deposition, drying time or raising as the cycle moves from the one compartment to the other.

Figure 18.1 – Tailings complex schematic

18.3.5 Infrastructure

- *Impoundment wall* – The tailings facility will consist out of an outer impounded wall. Constructed with suitable material and compacted in layers. It will be 20 m high and can be done in phases over the life period.
- *Spigot delivery system* – Ring feed delivery system in each compartment with 30 m take -off points controlled by its own valve system. This will assist to make sure that the pool is controlled around the penstock system.

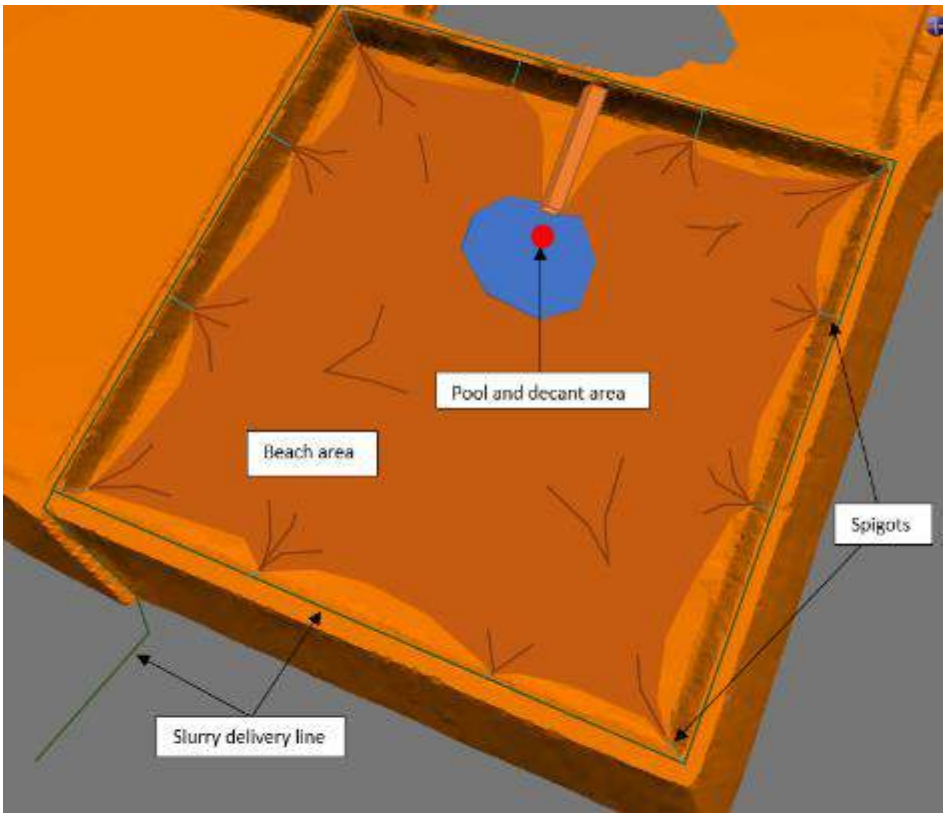


Figure 18.2 - Typical spigot deposition method

- *Filter and Blanket drain system* – a 6 m wide blanket drain will be installed around the inner toe of each compartment. This is to collect any seepage water and to control the phreatic level, to make sure it does not saturate the outer wall and causes stability concerns.

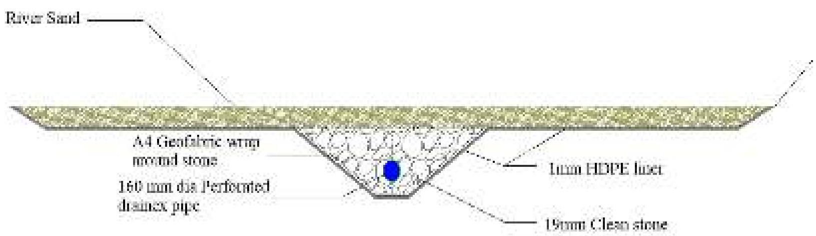


Figure 18.3 - Typical blanket drain

- *Decant system* – each compartment is equipped with a penstock decant system with a side and top intake. Precast penstock rings will be placed on top as the level rises.

- *Catchment paddocks and solution trench* – The outer toe area will be surrounded by catchment paddocks for any run-off or spillages. A lined solution trench will run on the outside perimeter to collect all seepage water from the drain system. This will be diverted and linked up to the current infrastructure.

18.3.6 Cost estimation

The cost estimation is based on the methodology and infrastructure under point 18.1 in this report. It can be summarized as the following in Table 18.1:

Table 18.1 - Tailings Cost Estimation Summary

No	Description	Amount (US$)
1	***Preliminary and General***	**$ 1 761 667**
2	***General Earthworks Tailings Dam***	$ 13 876 722
3	***Drains***	$ 2 066 986
4	***Solution dirty water trenches***	$ 217 934
5	***Fence***	$ 18 706
6	***Lights***	$ 68 438
7	***Penstock and inlets***	$ 547 728
8	***Reticulation***	$ 1 591 274
	Total	**$ 20 149 454**

The following assumptions have been made and are excluded from the cost estimation:

- Conveyor system on the coarse tailings dump is excluded. It is assumed that the existing system will be moved to the new area.
- All electricity work provided by the mine where required.
- The main feed delivery line and return water lines are connected to the existing infrastructure. The specific point was unknown during this investigation therefore the cost is based to the surroundings of the facility
- No DTM survey was available for the surrounding area. It is assumed that the area is relatively flat. The volumes and cost are based on this criterion.
- The current facilities do not have a liner system and is therefore not considered in the future facility as well.
- The provision above is only construction and excludes operating costs.

19 Market Studies and Contracts

Under the terms and conditions contained within ML2008/6L, Boteti will hold open tenders for sale of diamonds in Botswana. In the period 2012 to the end of 2014 dual viewing of goods were held in Antwerp and Gaborone with the final tender closing in Antwerp. Since January 2015 all diamond tender viewings and sales have taken place in Lucara's dedicated Sales and Marketing office within the Diamond Technology Park, Gaborone. Lucara manages a rough price book (>4000 price points) that generates a reserve price for each sales lot. Specials (+10.8ct and coloured diamonds) are treated on an individual basis. The Government Diamond Valuator ("GDV") also does a valuation of the rough lots to be tendered and reserve prices are compared prior to tender. The costs of the GDV are for the account of the Government. Royalty payments are calculated on the actual sales price for achieved during tenders.

19.1 Diamond sales

Since 2012 over 1.8 million carats of combine North, Centre and South lobe diamonds have been sold for revenue of US$1.1 billion (average price per carat of US$596/ct).

Sales lots are prepared for presentation to clients by Boteti Mining (Pty.) Ltd. staff in a modern, ultra-secure sorting facility. Sales parcels conform to industry standard size ranges and descriptions.

Karowe mine production includes on a consistent basis a proportion of large, high value Type IIa diamonds and more infrequently coloured diamonds (blue, pink, yellow). Diamonds such as these are very rare and command a special niche within the rough and polished markets. In 2013 Lucara implemented a tender sales mechanism referred to as "Exceptional Stone Tender" and has conducted 11 such tenders. Since 2013 the base value for a diamond to be included in an EST (total reserve value) has increased from US$250k to approximately US$1million. The terms and conditions of the exceptional stone tender are the same as regular tenders.

Timing of tender dates is aligned with other major southern African rough diamond sales dates to maximum participation of buyers. Sales are by closed tender with bidding conducted by an online platform. Results are announced at the close of the tender witnessed by a court appoint bailiff. Invoicing is immediate and payment is due in 5 business days, clients receive their winning parcel(s) once payment is received. Clients are required to register and undergo a verification process consisting of a variety of background checks including but not limited to proof of funds, Bourse membership, business trading license, and compliance to the Kimberley Process.

19.2 Client Base

Lucara has developed a strong, geographically diverse following of clients. Lucara has over 670 registered clients, including 161 new companies that registered in 2016, demonstrating a strong interest in the Karowe production. Attendance at tenders has increased to an average of 119 companies in the period of 2016-17 compared to 92 in 2015.

19.3 Rough Diamond Market Outlook

The overall rough and polished markets remain cautious as the supply and demand fundamentals remain unbalanced. New rough producers came online through 2016 and 2017 (Renard, Gahcho Kue, Liqhobong) achieved market prices for the new productions which have not met expectations but are improving.

Demonitization in India had an overall impact on the market but in terms of rough pricing the impact was not as significant with prices off mainly in poorer quality smaller goods. Large volumes of rough continue to be sold by the majors (De Beers, Alrosa) with a strong rebound in the price of rough in the categories where demand was affected the greatest by the demonetization in India in November 2016. Although polished diamond sales lagged and in general decreased the market for rough diamond sales remained robust through late 2016 and early 2017 based on available liquidity and year to date rough sales. Lucara is advantageously placed in the market with the high value large diamonds, and this market remains robust due to lower than historical large stone recoveries by other producers. Demand for Karowe large diamonds remained strong in HY1 2017. The average prices in the Q1 2017 tender were amongst the top three in terms of US$/ct achieved over the 30 Lucara regular tenders held to date.

A strong customer base which is expanding, excellent participation in Tenders and a consistent sorting and presentation of sales lots has generated a Lucara brand where the outlook is positive.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies Completed to Date

Two pre-mining environmental studies were conducted for the Karowe Mine (formerly known as the AK6 project), namely an Environmental Impact Assessment ("EIA") Study for AK6 (Geoflux, 2007) and Environmental Management Plan (EMP) for the AK6 Diamond Mine (SiVEST, 2010). The Botswana Department of Environmental Affairs approved both studies in 2008 and 2010, respectively. In terms of the Mining License ((ML2008/6L) Boteti Mining was granted common law surface rights over the entire mining license area and the access road for the duration of the mining lease. The mine was commissioned in October 2011 with the commissioning of the processing facilities commencing in April 2012.

The Initial EIA (Geoflux, 2007) was granted with conditions, all of which KDM, in the opinion of previous QPs evaluating the operation, met or continues to meet. Subsequent to this, the EMP was updated in 2013 and again in 2016 to comply with the requirements of Botswana's evolving environmental legislation, notably the Environmental Assessment Act of 2011, and to assess the activities and associated impacts of the expansion of the process plant and the Bulk Sampling Plant (Geoflux 2016). As part of this process, KDM also received approval for its Archaeological Clearance Certificate (EBS, 2012) as well as the Water Rights for its groundwater abstraction and monitoring boreholes (Geoflux, 2016). The Water Rights were granted in 2008, 2010, 2011 and 2014.

Permitting applications for the site's waste facilities (salvage yard, landfill, sewage plant and incinerator) initiated over the past three years, remain in process (Geoflux, 2016).

KDM has developed a legal register which is used to track legal changes as they apply to the operation and its activities (EBS, 2017).

20.2 Environmental Management

As required in terms of the Environmental Assessment Act of 2011, the 2016 EMP update sets out the mitigation measures and impact management / monitoring activities that KDM should undertake to maintain compliance during the operational and later the closure phase of the project. Specifically the mine monitors:

- Air quality (dust).
- Groundwater Quality.
- Waste management.
- Environmental Incidents.

KDM also conducts a series of regular activities in terms of the following actions plans:

- Biodiversity Action Plan.
- Health and Safety Plan.
- Groundwater Control Plan.
- CSI and Labour Plan.
- Heritage Plan.
- Stakeholder Engagement Plan.
- Grievance Response Procedure.
- Emergency Response Plan.

- Community Health Safety and Security Management Plan.
- KDM Waste Management Plan.

As incidents occur they are logged, addressed and closed out. Where monitoring results indicate the need for corrective actions, these are developed and implemented over time. Various reviews have recommended improvements in data gathering processes (Geoflux 2014/ EBS 2017) and the 2016 Assurance process has highlighted “the need for improved data quality controls.”

20.3 Water Usage and Management

Groundwater is the primary source of water for various uses throughout the Boteti sub-district, which includes other diamond mines, village water supply and agriculture. The regional water quality is brackish. Pre-mining groundwater baseline information was collected in 2010 and used to construct a conceptual mine water balance. The mine has an abstraction permit for 8 million cubic metres p.a. with annual abstraction over the past 5 years around 2.5 million cubic meters. (Geoflux 2016) Approximately 20% of water needs are met from water recovery from the slimes dam’s return water dam. No water is discharged from site.

Due to the need for pit dewatering, there is localised groundwater level depression around the mine. (Geoflux 2016) The Groundwater modelling has shown that the abstraction rates required for pit dewatering and the wellfield to meet the long term water demand will lower the regional water levels and several farmers in the vicinity may be progressively affected. This effect could be amplified by cumulative impacts arising over time as KDM and other regional water users, such as nearby diamond operations, continually draw on the Ntane aquifer which forms the principal regional water source.

A 2012 due diligence found that the groundwater model required updating and that the mine water balance remained conceptual and required calibration according to actual process data. This was confirmed by a 2016 ESG Compliance Assessment, which also recommended improvements to the storm water management infrastructure associated with the site’s waste facilities (salvage yard, landfill, and sewage plant). The development of a water management computer model was commissioned for 2016.

20.4 Slimes Dam

The square-shaped slimes dam is located south of the open pit. The slimes dam is split into four equal sized compartments with a total footprint of approximately 146 ha, which are operated on a rotational basis (approximately three continuous months per annum for each) in order to minimize water losses. As stipulated in the EMP, seepage run-off and dust fallout from the dump are monitored on an on-going basis.

20.5 Waste Rock Dump

The square-shaped waste rock dump is located west of the slimes dam and accommodates all waste rock not used for slimes dam impoundment construction. The footprint of the waste rock dump is approximately 100 ha. The waste rock dumps side slopes will be constructed to a gradient of 1:3 and the maximum vertical height of the waste rock dump will be 25 m.

As stipulated in the EMP, seepage run-off and dust fallout from the dump are monitored on an on-going basis.

20.6 Social and Community

Karowe has developed and implemented a formal Stakeholder Engagement Plan which includes a Grievance Resolution Procedure. Stakeholder meetings take place on a quarterly basis. The key stakeholder concern is over groundwater resources which communities believe are declining due to Karowe's abstraction. They look to the mine to address issues of potential loss of groundwater access or usage. (Geoflux 2016)

A Community Social Responsibility (CSR) programme has been developed and implemented with focus on entrepreneurship development and support, local community infrastructure, protection of vulnerable groups, and wildlife conservation. Beyond this programme there are no material commitments to which Karowe needs to deliver.

20.7 Mine Closure

A mine closure and rehabilitation plan is a requirement under Section 65 of the Botswana Mines and Minerals Act (1999), under which Boteti Mining (Pty) Ltd is obliged, to develop and implement a mine closure plan during the Life of Mine and to ensure that the mining lease area is progressively rehabilitated and ultimately reclaimed at the end of life of mine to the satisfaction of the Director of Mines.

The company makes restoration provisions for the eventual closure and rehabilitation of Karowe. A conceptual mine closure plan for Karowe was incorporated into the pre-mining EIA (approved 2008) and the EMP submitted and approved in 2010 following Lucara's takeover of the then AK6 Diamond Mine project. This was expanded into a high-level cost estimate in 2011 when the operation was commissioned. In 2013, KDM commissioned Geoflux to develop a Mine Closure and Rehabilitation Plan (MCRP) based upon site survey information (Geoflux 2013). As a result of this a Financial Guarantee was raised by KDM with Standard Chartered (SC) for BWP 100 million in June 2014.

The detailed MCRP set out site closure options, objectives and criteria for both unscheduled and scheduled closure, calculating BWP 41.3 million (US$4.67 million) and BWP 123.6 million (US$13.89 million) respectively (Geoflux 2016). The Botswana Department of Mines has commented on the MCRP but not yet formally approved it.

Concurrent rehabilitation takes place at the exploration sites, but rehabilitation at Karowe is not scheduled to commence before 2022. The closure liability calculation is based on annually updated master rates used for closure planning in South Africa. As is common practice in southern African mining operations at this stage of mining, the cost for water treatment is excluded due to insufficient information on future groundwater impacts and potential treatment costs. Based on the experience of other Botswana diamond mines, it is unlikely that material mine decant will occur during the closure process.

Following increases in the tailings dump and stockpile areas the provisions, as at December 2016, stand at an un-discounted US$19.4m (AFS, 2016). The current closure plan considers all closure liabilities up to December 2016. As the mining operation and Botswana mine closure guidance evolve, the closure liability estimates will require further refinement.

20.8 Recommendations

For the PFS, a more detailed assessment will be required to determine the potential need for any additional land. Assuming no additional land is required for the underground expansion, the existing Environmental Impact Assessment (EIA) will require updating as part of Lucara's regular regulator and stakeholder engagement process. This would include an updating of the mine closure liability estimate.

Should additional land be required for the underground expansion, the Project Mining Lease and its EIA would require public scoping and review. Using its existing Stakeholder and regulator Engagement processes, the mine would be required to disclose the objectives of the expansion as well as any new impacts that may be associated with the underground project implementation, including a general project schedule. More detailed investigations in terms of the

EIA update would include the development of a formal, quantitative and qualitative surface and groundwater model for the project, a recalculation of the mine closure liability, and targeted investigations into new opportunities for socio-economic project and commercial business development which would result in broader socio-economic benefits and thereby strengthen the project's social license to operate.

21 Capital and Operating Costs

21.1 Capital Cost Estimates

The capital cost estimates were based on budget pricing from suppliers for high cost items such as underground mining development. Benchmarked costs from similar projects were also factored based on industry norms for the typical facilities required for the underground operation. Labour rates were based on actual cost to company rates at the Karowe mine.

Table 21.1 below shows the summary of the estimated capital required.

Table 21.1 – Summary of Capital Costs

Area	Capital (2018-2025) (US$ '000)	Capital (2026-2037) (US$ '000)	Area Totals (US$ '000)
Development Capital	$86 256	$77 416	$163 672
Engineering Capital	$48 530	$14 874	$63 404
Tailings storage facility	$20 920	$6 781	$27 701
Closure Costs	$0	$20 000	$20 000
Capital Provisions	$0	$27 000	$27 000
Net Total Capital	$155 706	$146 071	$301 777
Contingency	$38 926	$31 418	$70 344
Total Capital	**$194 632**	**$177 489**	**$372 121**

The following is noted on the items in Table 21.1:

- Development capital includes the procurement of the development equipment, the development capital required for a twin decline system, the associated ramps, rim tunnels and access cross cuts.
- Engineering capital includes the estimated costs for additional surface infrastructure to support the underground mine requirements whilst making use of the existing surface infrastructure, electrical supply upgrade, underground pump stations and project development costs.
- Tailings storage facility estimate is for the expansion of the existing facility to cater for the underground requirements.
- Closure costs is the provision for the closure in accordance with the applicable legislation at the end of the underground operations.
- Capital provisions included the annual working capital requirements.
- A 25% contingency was applied to all capital expenditure, to account for the expected difficulty of developing through the host rock. Contingency is excluded for the last year of life of mine.

Table 21.2 shows the estimated capital requirements by year over the life of mine. The tailings storage facility and the closure costs are included under engineering capital in this table.

Table 21.3 shows the engineering capital requirements by year.

Table 21.2 – Capital Expenditure by Year

Description	US$	Total	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Upfront Mining Capital	$('000)	**86 256**			9 631	7 581	18 153	15 399	14 163	21 329												
Ongoing Mining Capital	$('000)	**77 416**									15 250	13 048	9 086	8 478	7 176	6 787	6 138	5 952	2 366	2 366	769	
Capital Provisions	$('000)	**27 000**										3 000	3 000	3 000	3 000	3 000	3 000	3 000	3 000	3 000		
Engineering Capital	$('000)	**111 105**	16 660	6 086	4 527	15 280	14 726	1 649	31	10 491	10 691	779	779	31	31	608	7 560	779			10 200	10 200
Contingency	25%	**70 344**	4 165	1 522	3 540	5 715	8 220	4 262	3 548	7 955	6 485	4 207	3 216	2 877	2 552	2 599	4 174	2 433	1 342	1 342	192	
TOTAL CAPITAL	**$('000)**	**372 121**	20 825	7 608	17 698	28 577	41 098	21 310	17 742	39 774	32 426	21 033	16 081	14 385	12 759	12 993	20 872	12 163	6 708	6 708	11 162	10 200

Table 21.3 – Summary of Engineering Capital by year

WBS	Area	Total (US$)	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
100	**Surface Infrastructure**	**27 106 341**				**13 264 709**	**13 264 709**									**576 923**						
	110 -Electrical supply - Upgrade Transformers	576 923														576 923						
	110 - Roads and Terracing	4 284 659				2 142 330	2 142 330															
	120 - Fuel and Lubricants	1 098 065				549 033	549 033															
	130 - General Infrastructure/ Buildings	7 422 792				3 711 396	3 711 396															
	140 - Surface Water Reticulation	13 723 902				6 861 951	6 861 951															
300	**North Decline - Infrastructure**	**2 255 259**				**121 150**	**493 920**	**372 770**				**316 855**	**316 855**				**316 855**	**316 855**				
	320 - Pumping																					
	321 - Pump station 1	745 540					372 770	372 770														
	322 – Pump station 2	633 709										316 855	316 855									
	323 – Pump station 3	633 709															316 855	316 855				
	325 - Temporary Dewatering	242 301				121 150	121 150															
400	**South Decline - Infrastructure**	**2 255 259**				**121 150**	**493 920**	**372 770**				**316 855**	**316 855**				**316 855**	**316 855**				
	420 - Pumping																					
	421 – Pump station 1	745 540					372 770	372 770														
	422 – Pump station 2	633 709										316 855	316 855									
	423 – Pump station 3	633 709															316 855	316 855				
	425 - Temporary Dewatering	242 301				121 150	121 150															
500	**Shaft Electrical**	**2 929 202**				**788 616**	**473 196**	**903 428**	**30 769**	**30 769**	**30 769**	**144 837**	**144 837**	**30 769**	**30 769**	**30 769**	**144 837**	**144 837**				
	Electrical	2 117 309				745 001	295 385	769 231	30 769	30 769	30 769	30 769	30 769	30 769	30 769	30 769	30 769	30 769				
	Pump stations - electrical	811 893				43 614	177 811	134 197				114 068	114 068				114 068	114 068				
	Mining contractor Mob/Demob Provision	**600 000**									200 000										200 000	200 000
600	**Other**	**75 958 791**	**16 660 288**	**6 086 049**	**5 413 310**	**98 446**				**10 459 811**	**10 459 811**						**6 781 077**				**10 000 000**	**10 000 000**
	Drilling	9 000 000	4 500 000	4 500 000																		
	Feasibility	1 944 623		486 156	1 458 467																	
	Geotech	4 143 982	4 143 982																			
	FEED	2 953 396			2 854 950	98 446																
	Owners Costs	2 841 290	641 504	1 099 893	1 099 893																	
	Tailing Dam - Phase 1	20 919 622								10 459 811	10 459 811											
	Tailing Dam - Phase 2	6 781 077															6 781 077					
	Dewatering - Program A	4 880 043	4 880 043																			
	Dewatering - Program B	2 494 760	2 494 760																			
	Closure Cost	20 000 000																			10 000 000	10 000 000
Total	**US$**	**111 104 852**	**16 660 288**	**6 086 049**	**5 413 310**	**14 394 071**	**14 725 746**	**1 648 968**	**30 769**	**10 490 580**	**10 690 580**	**778 546**	**778 546**	**30 769**	**30 769**	**607 692**	**7 559 623**	**778 546**			**10 200 000**	**10 200 000**

Figure 21.1 shows a graph of the capital spend for the underground mine up to full production.

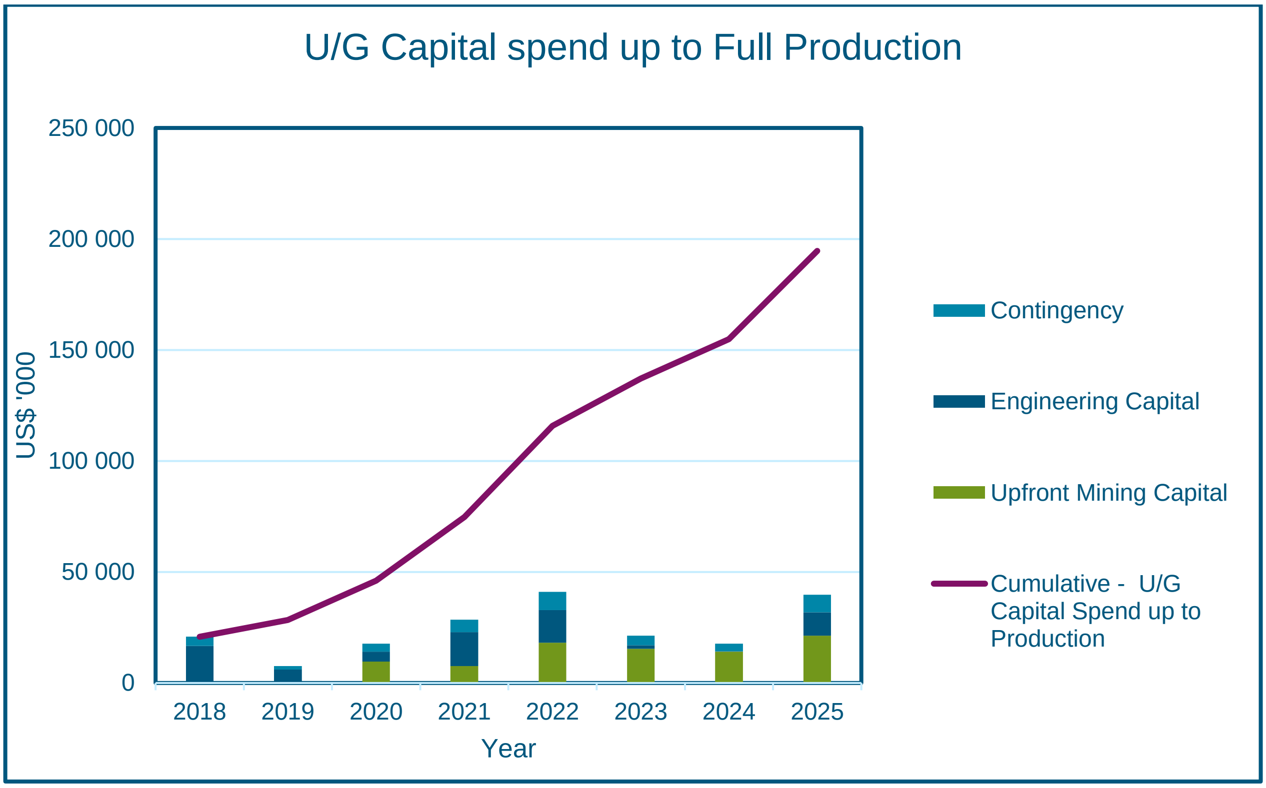


Figure 21.1 – Underground capital spend up to full Production

21.1.1 Key capital cost estimation parameters

The following key estimate parameters were applied to the capital cost estimate:

- Estimate Class: The capital cost estimates are at a preliminary economic assessment level of ±35%.
- Estimate Base Date: The base date of the estimate is September 2017.
- Currency: All capital costs are expressed in United States Dollars (US$). Portions of the estimate were estimated in South African Rand (ZAR) and Botswana Pula (BWP) and converted to United States Dollars. Exchange rates used, stipulated by the Client, were:
  - ZAR : US$ = 13 : 1
  - BWP : US$ = 10 : 1

21.1.2 Basis of capital estimate

Underground mine development costs were estimated on a per meter basis of development. Different rates were used for the different host rock including:

- Sandstone and granite.
- Red mudstone.
- Mudstone.
- Kimberlite.

The estimate was based on the following variable inputs for the different host rock:

- Labour.
- Fuel.

- Oil.
- Maintenance.
- Tyres.
- Support required.
- Drilling.
- Blasting.
- Equipping.

The fixed cost elements included:

- Mobile equipment including drill rigs, loaders, scaler, bolter, haul trucks, utility vehicles and transporters.
- Fixed equipment including fans and associated electrical infrastructure.
- Installations including water handling, service bays, refuelling, stores and explosives.
- Personal equipment including cap lamps, self-rescuers and gas monitoring requirements.

The costs were developed from benchmarked database information.

Infrastructure costs were based on in-house database costs and scaled for the size and quantity of infrastructure required.

21.1.3 Capital cost estimate exclusions

The following items were excluded from the capital cost estimate:

- Financing costs.
- Currency fluctuations.
- Lost time due to severe weather conditions beyond those expected in the region.
- Lost time due to force majeure.
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule.
- Stores inventories, other than those supplied in initial fills, capital spares, or commissioning spares.
- Any project sunk costs (current studies, exploration programs, etc.).
- Escalation cost.

21.2 Operating Costs

The operating cost estimate was based on a combination of experience, reference projects, benchmarked operating costs, budgetary quotes and factoring as appropriate with a preliminary study.

The operating cost estimate includes the costs to mine underground from 2023 ramping up to full production in 2026. The estimate also included costs for processing the kimberlite and the associated engineering support costs. The target accuracy of the operating cost estimate was ±35%. Production through the process plant (milled tonnage) is kept consistent by supplementing tonnage from the stockpiles.

A summary of the operating costs is shown in Table 21.4. The costs are in United Stated Dollars (US$). The table shows the life of mine operating cost for the underground mine. A 10% contingency is included in the costs.

Table 21.4 – Summary of Operating Costs

Description	Total Operating Costs LOM ($000's)	Average Cost per Tonne ($/t processed)
Mining*	766 532	31.00
Processing	292 271	11.82
Engineering	50 786	2.05
Contingency (10%)**	110 959	4.49
Subtotal	**$1 220 548**	**$49.36**
Sustaining Development Capital	119 198	4.82
All-in Sustaining Costs	**$1 339 746**	**$54.18**

**Note that the underground operating costs will commence in 2023*

*** Contingency of 10% applied only to underground mining, processing and engineering*

Table 21.5 shows a breakdown of the annual operating costs per year.

Description	US$	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Mining U/G	$('000)	766 532	1 158	14 170	38 505	77 127	77 115	76 624	76 125	76 102	75 259	77 150	76 279	70 450	29 777	691
Processing Cost	$('000)	292 271	442	5 403	14 682	29 408	29 403	29 216	29 026	29 017	28 696	29 416	29 084	26 862	11 354	263
Engineering Cost UG	$('000)	50 786	3 330	3 330	3 330	3 330	3 330	3 346	3 346	3 346	3 346	4 138	4 153	4 153	4 153	4 153
Contingency OPEX - U/G	10%	110 959	493	2 290	5 652	10 987	10 985	10 919	10 850	10 847	10 730	11 070	10 952	10 146	4 528	511
TOTAL OPERATING COST		**1 220 548**	**5 423**	**25 194**	**62 169**	**120 852**	**120 834**	**120 104**	**119 347**	**119 312**	**118 031**	**121 774**	**120 468**	**111 611**	**49 812**	**5 618**

Table 21.5 – Summary of Annual Operating Costs

21.2.1 Basis for operating cost estimates

The underground mining operating cost was based on benchmarking similar operations. A breakdown of the mining operating cost is shown in Table 21.6.

Table 21.6 – Underground Mining Operating Cost

Description	Cost per Tonne (US$)
Development Cost	$ 12.79
Drilling	$ 2.15
Blasting	$ 0.42
Load and Haul	$ 2.86
Slot Cutting	$ 0.37
Engineering Support	$ 3.26
Secondary Breaking	$ 0.54
Primary Crushing	$ 0.20
Ground Handling	$ 4.24
Roadways	$ 0.34
Direct Mining Cost	**$ 14.37**
Mineral Resource Mgt	$ 1.60
Mining Systems	$ 0.03
Mining Overheads	$ 2.13
Direct Overheads	**$ 3.77**
Mining Total	**$ 31**

- Underground water pumping costs were based on database costs and include provision for maintenance parts, labour and electrical costs for the various systems.
- Ventilation costs were based on estimated electricity usage based on the fan sizes required for ventilation purposes. Electricity costs were based on the actual electricity price of US$0.05145/kWh charged to the mine.
- Process operating costs were based on actual operating costs at Karowe mine and these costs are expected to remain the same as the tonnage profile for the mine remains consistent.
- Labour costs were based on actual total cost to company rates currently applicable to Karowe mine.

22 Economic Analysis

22.1 Summary of Results

A preliminary economic model was developed based on the estimated capital and operating costs summarised in Section 21.

It must be noted that this economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable the mining target to be categorized as mineral reserves. As such, the outcome of the PEA is indicative only.

22.2 Key Assumptions and Input Parameters

Table 22.1 outlines the key assumptions and input parameters used in the economic analysis.

Table 22.1 – Key Economic Assumptions and Model Input Parameters

PARAMETERS	VALUE
Exchange Rate (ZAR/US$)	13.00
Exchange Rate (ZAR/P)	1.30
Exchange Rate (P/$)	10.00
Marketing cost ($ '000) pa	2 918
Site Admin ($'000) pa	8 902
Mine Call Factor	92%
Corporate Overhead Cost ($ ‘000) pa	3 960
Minimum Tax Rate	22%
Contingency Operating Cost	10%
Contingency Capital Cost	25%
Discount Factor	8%
Royalties	10%
Closure Cost ($ '000)	20 000
Rough diamond price – South lobe (2017) (US$/carat)	$730
Rough Diamond Annual Real Diamond Price Escalator (%)	2.5

The diamond price for the south lobe based on the 2017 average price model is escalated by 2.5% per annum over the life of the mining schedule proposed in the PEA. The marketing cost, site administration cost and corporate overhead costs were based on actual costs at Karowe mine. No project financing cost has been included.

22.3 Mine Production

A summary of the mine production schedule is shown in Table 22.2 below. The annual production milled remains constant over the life of mine at 2 605 000 tonnes per annum.

Table 22.2 – Summary of the Mine Production Schedule

Area	Rate	TOTAL	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
TOTAL ROM TONNES MINED	Tonnes	44 901 469	3 119 995	2 898 859	2 762 170	2 762 170	2 762 078	2 577 533	2 947 280	2 081 121	2 487 983	2 487 589	2 471 729	2 455 653	2 454 909	2 427 720	2 488 695	2 460 606	2 272 565	960 535	22 281	
TONNES MILLED	tonnes	50 780 324	2 602 950	2 605 670	2 605 752	2 605 752	2 605 752	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	2 605 670	1 275 069
DIAMONDS PRODUCED	ct	5 856 905	288 385	318 699	374 430	315 911	336 007	325 884	339 634	312 063	314 608	340 181	326 627	305 458	270 358	305 005	311 646	295 138	291 128	173 011	193 579	119 154
Selling Price	$/ct	884.16	718.38	691.11	640.88	749.21	726.96	790.84	765.82	802.20	889.36	897.04	937.09	960.85	981.68	1 006.22	1 031.38	1 057.16	1 083.59	1 117.92	1 155.97	1 166.91

Figure 22.1 shows the contribution of revenue from both the open pit and underground per year.

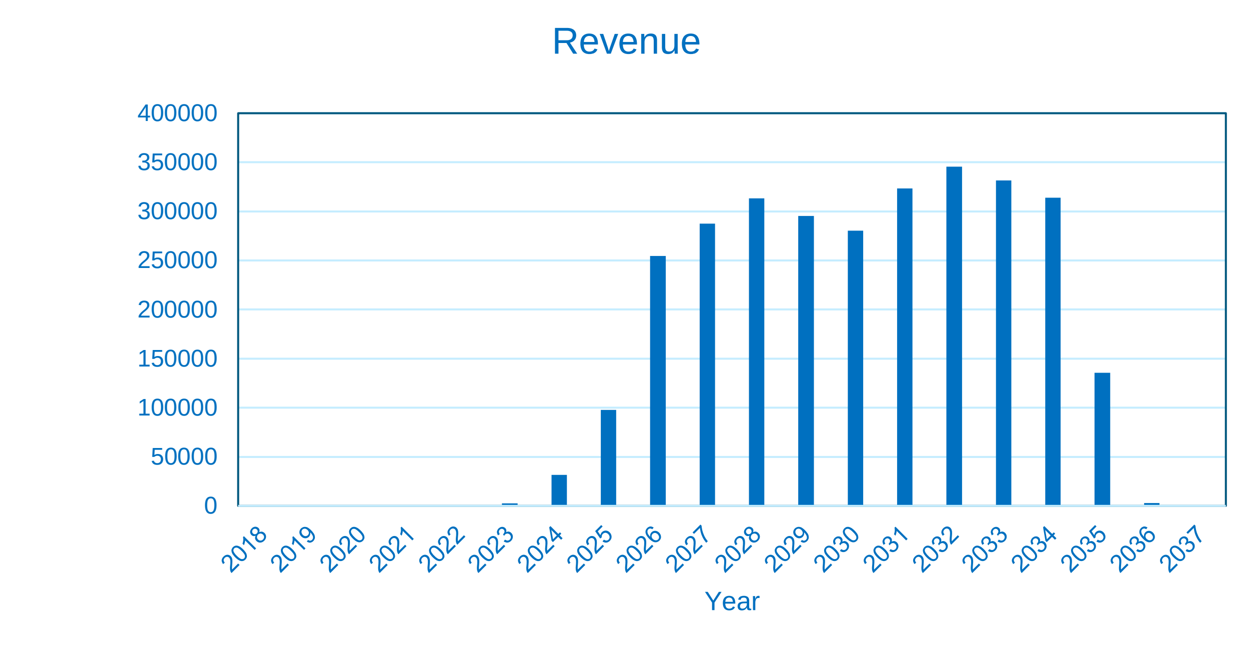


Figure 22.1 – Figure showing revenue contribution from open pit and underground operations

Figure 22.2 shows the cashflow after taxation by year.

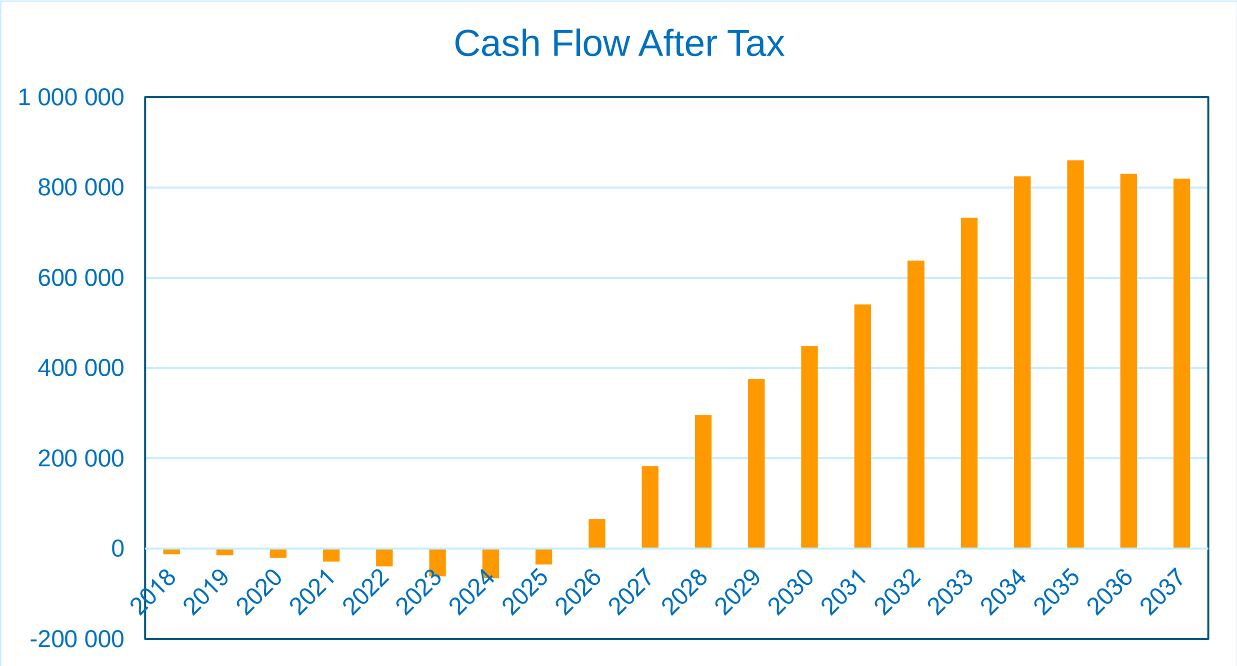


Figure 22.2 – Cashflow after Taxation

Table 22.3 shows the summary of the financial model.

Table 22.3 – Summary of Financial Model

UNDERGROUND PROJECT - FINANCIAL MODEL US$ ('000)																								
Capital and Operating Costs																								
Item	Area	Rate	TOTAL	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Statistics	TOTAL ROM TONNES MINED	tonnes	24 726 833						37 357	457 104	1 242 109	2 487 983	2 487 589	2 471 729	2 455 653	2 454 909	2 427 720	2 488 695	2 460 606	2 272 565	960 535	22 281		
	TONNES MILLED	tonnes	24 726 833						37 357	457 104	1 242 109	2 487 983	2 487 589	2 471 729	2 455 653	2 454 909	2 427 720	2 488 695	2 460 606	2 272 565	960 535	22 281		
	DIAMONDS PRODUCED	ct	2 993 728						2 867	36 548	109 875	279 134	307 826	327 194	300 984	278 553	313 452	326 863	306 068	282 608	119 173	2 582		
	Selling Price	$/ct	1007.52					825.85	846.54	867.73	889.43	911.67	934.46	957.83	981.76	1006.32	1031.48	1057.26	1083.69	1110.78	1138.55	1110.78	1138.55	
	Selling Price	P/ct	10 075.23					8 259	8 465	8 677	8 894	9 117	9 345	9 578	9 818	10 063	10 315	10 573	10 837	11 108	11 386	11 108	11 386	
	Recovered Grade	ct/ht	12.11																					
Capital cost	Upfront Mining Capital	$('000)	86 256			9 631	7 581	18 153	15 399	14 163	21 329													
	Ongoing Mining Capital	$('000)	77 416									15 250	13 048	9 086	8 478	7 176	6 787	6 138	5 952	2 366	2 366	769		
	Capital Provisions	$('000)	27 000										3 000	3 000	3 000	3 000	3 000	3 000	3 000	3 000	3 000			
	Engineering Capital	$('000)	111 105	16 660	6 086	4 527	15 280	14 726	1 649	31	10 491	10 691	779	779	31	31	608	7 560	779			10 200	10 200	
	Contingency	25%	70 344	4 165	1 522	3 540	5 715	8 220	4 262	3 548	7 955	6 485	4 207	3 216	2 877	2 552	2 599	4 174	2 433	1 342	1 342	192		
	TOTAL CAPITAL		**372 121**	**20 825**	**7 608**	**17 698**	**28 577**	**41 098**	**21 310**	**17 742**	**39 774**	**32 426**	**21 033**	**16 081**	**14 385**	**12 759**	**12 993**	**20 872**	**12 163**	**6 708**	**6 708**	**11 162**	**10 200**	
Operating cost	Mining U/G	$('000)	766 532						1 158	14 170	38 505	77 127	77 115	76 624	76 125	76 102	75 259	77 150	76 279	70 450	29 777	691		
	Processing Cost	$('000)	292 271						442	5 403	14 682	29 408	29 403	29 216	29 026	29 017	28 696	29 416	29 084	26 862	11 354	263		
	Engineering Cost UG	$('000)	50 786						3 330	3 330	3 330	3 330	3 330	3 346	3 346	3 346	3 346	4 138	4 153	4 153	4 153	4 153		
	Contingency OPEX - U/G	10%	110 959						493	2 290	5 652	10 987	10 985	10 919	10 850	10 847	10 730	11 070	10 952	10 146	4 528	511		
	TOTAL OPERATING COST		**1 220 548**						**5 423**	**25 194**	**62 169**	**120 852**	**120 834**	**120 104**	**119 347**	**119 312**	**118 031**	**121 774**	**120 468**	**111 611**	**49 812**	**5 618**		
CASH FLOW																								
Item	Area	Rate	TOTAL	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
	Revenue	$('000)	3 016 248						2 427	31 714	97 727	254 479	287 651	313 395	295 495	280 314	323 319	345 578	331 682	313 915	135 685	2 868		
	Operating Cost	$('000)	1 220 548						5 423	25 194	62 169	120 852	120 834	120 104	119 347	119 312	118 031	121 774	120 468	111 611	49 812	5 618		
	Overhead Cost -Site Admin	$('000)	124 623						8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902	8 902		
	Corporate O/H	$('000)	55 438						3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960	3 960		
	Royalty	$('000)	301 625						243	3 171	9 773	25 448	28 765	31 340	29 550	28 031	32 332	34 558	33 168	31 392	13 568	287		
	Marketing cost	$('000)	40 857						2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918	2 918		
	Profit / Loss (EBITDA)	$('000)	1 273 157						-19 018	-12 431	10 005	92 399	122 273	146 172	130 819	117 191	157 176	173 467	162 265	155 133	56 524	-18 817		
	Cum Profit/Loss (EBITA)	$('000)	1 273 157						-19 018	-31 449	-21 444	70 955	193 227	339 399	470 218	587 409	744 585	918 051	1 080 317	1 235 450	1 291 974	1 273 157	1 273 157	1 273 157
	Capital Expenditure	$('000)	372 121	20 825	7 608	17 698	28 577	41 098	21 310	17 742	39 774	32 426	21 033	16 081	14 385	12 759	12 993	20 872	12 163	6 708	6 708	11 162	10 200	
	Cash Flow Before Tax	$('000)	901 036	-20 825	-7 608	-17 698	-28 577	-41 098	-40 329	-30 173	-29 769	59 973	101 240	130 091	116 434	104 432	144 183	152 595	150 103	148 425	49 817	-29 979	-10 200	

Taxation	$('000)	318 843											16 550	37 179	31 055	52 430	54 979	55 320	56 810	14 519			
Net Taxation Benefit	$('000)	237 602	8 424	5 325	12 388	20 004	30 431	18 373	25 895	60 055	40 598	16 108											
Cash Flow After Taxation	$('000)	819 795	-12 401	-2 282	-5 309	-8 573	-10 667	-21 956	-4 278	30 286	100 571	117 348	113 541	79 254	73 377	91 753	97 615	94 783	91 615	35 298	-29 979	-10 200	
Cumulative Cash Flow after tax		819 795	-12 401	-14 684	-19 993	-28 566	-39 233	-61 189	-65 467	-35 181	65 391	182 739	296 280	375 534	448 911	540 663	638 278	733 061	824 676	859 974	829 995	819 795	819 795

TAXATION

	TOTAL	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Unredeemed Capex Balance brought forward			20 825	28 433	46 131	74 707	115 806	156 134	186 307	216 076	156 103	54 863									29 979	40 179
Capital Expenditure in year	372 121	20 825	7 608	17 698	28 577	41 098	21 310	17 742	39 774	32 426	21 033	16 081	14 385	12 759	12 993	20 872	12 163	6 708	6 708	11 162	10 200	
Total Capital Redemption		20 825	28 433	46 131	74 707	115 806	137 116	173 877	226 081	248 502	177 136	70 944	14 385	12 759	12 993	20 872	12 163	6 708	6 708	11 162	40 179	40 179
Profit / Loss (EBITDA)							-19 018	-12 431	10 005	92 399	122 273	146 172	130 819	117 191	157 176	173 467	162 265	155 133	56 524	-18 817		
Taxable Profit - After Capital Expenditure												75 228	116 434	104 432	144 183	152 595	150 103	148 425	49 817			
Redemption Carried Forward		20 825	28 433	46 131	74 707	115 806	156 134	186 307	216 076	156 103	54 863									29 979	40 179	40 179
Tax Rate												22.00%	31.93%	29.74%	36.36%	36.03%	36.85%	38.28%	29.14%			

Figure 22.3 shows a waterfall chart showing the various contributors of the underground project.

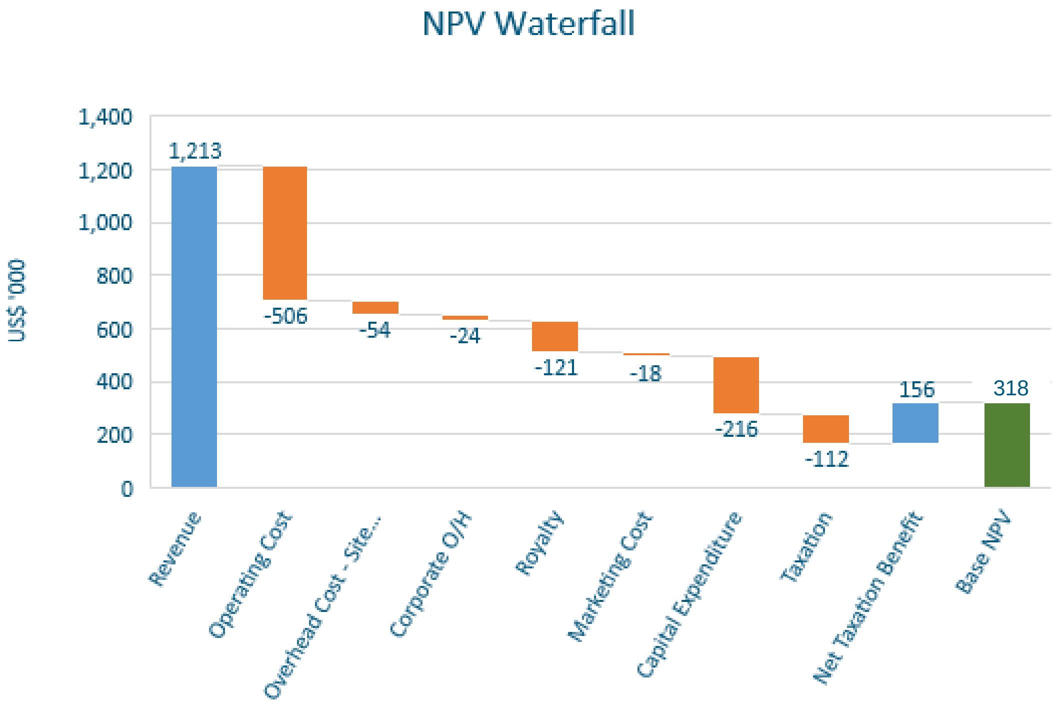


Figure 22.3 – NPV Waterfall chart of various contributors

22.4 Taxes, Royalties and Other Interests

The Karowe Mine is taxed according to a prescribed schedule of the Botswana Income Tax Act. Profits from the Karowe Mine are taxed according to the annual tax rate formula as follows:

> 70-(1500/x) where x is the profitability ratio given by taxable income as a percentage of gross income (provided that the tax rate will not be less than the minimum company rate). Boteti can offset capital expenditure against profit in the calculation of the variable Income Tax liability.

A royalty of 10% on actual sales of diamonds is levied by the Government of Botswana.

22.5 Sensitivity Analysis

Table 22.4 shows a table of the NPV at various discount rates.

Table 22.4 – Summary of NPV Sensitivity

NPV (2018 Basis) SENSITIVITY		
DISCOUNT RATE	US$ Millions	
5%	US$	451
8%	US$	318
11%	US$	226

Figure 22.4 shows the NPV sensitivity vs. Revenue, Operating Cost, and Capital expenditure variations of 10 percent.

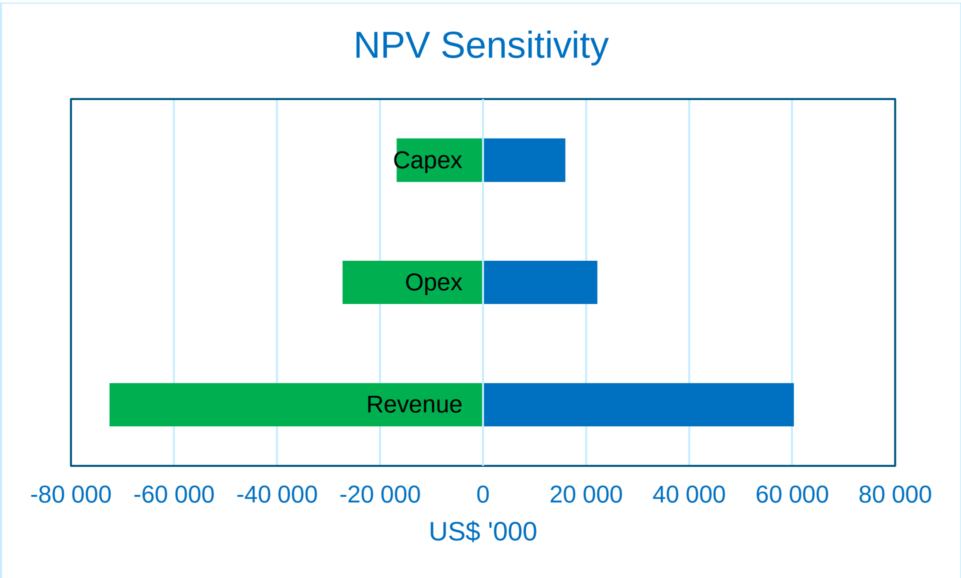


Figure 22.4 – NPV Sensitivity

22.6 Results

The result of the PEA financial model shows a positive after-tax NPV of US$451 Million at a 5% discount rate, and a free cash flow of US$820 Million after taxes.

The Internal Rate or Return (IRR) of the project is 38.9% and payback period (pre-tax) of 2.5 years.

23 Adjacent Properties

The Karowe Mine is based on the AK6 kimberlite pipe, which is part of the Orapa kimberlite field. A total of nine kimberlite pipes in this field are either operating mines, or have recently been mined. The primary adjacent diamond mining properties are shown in Figure 23.1 below.

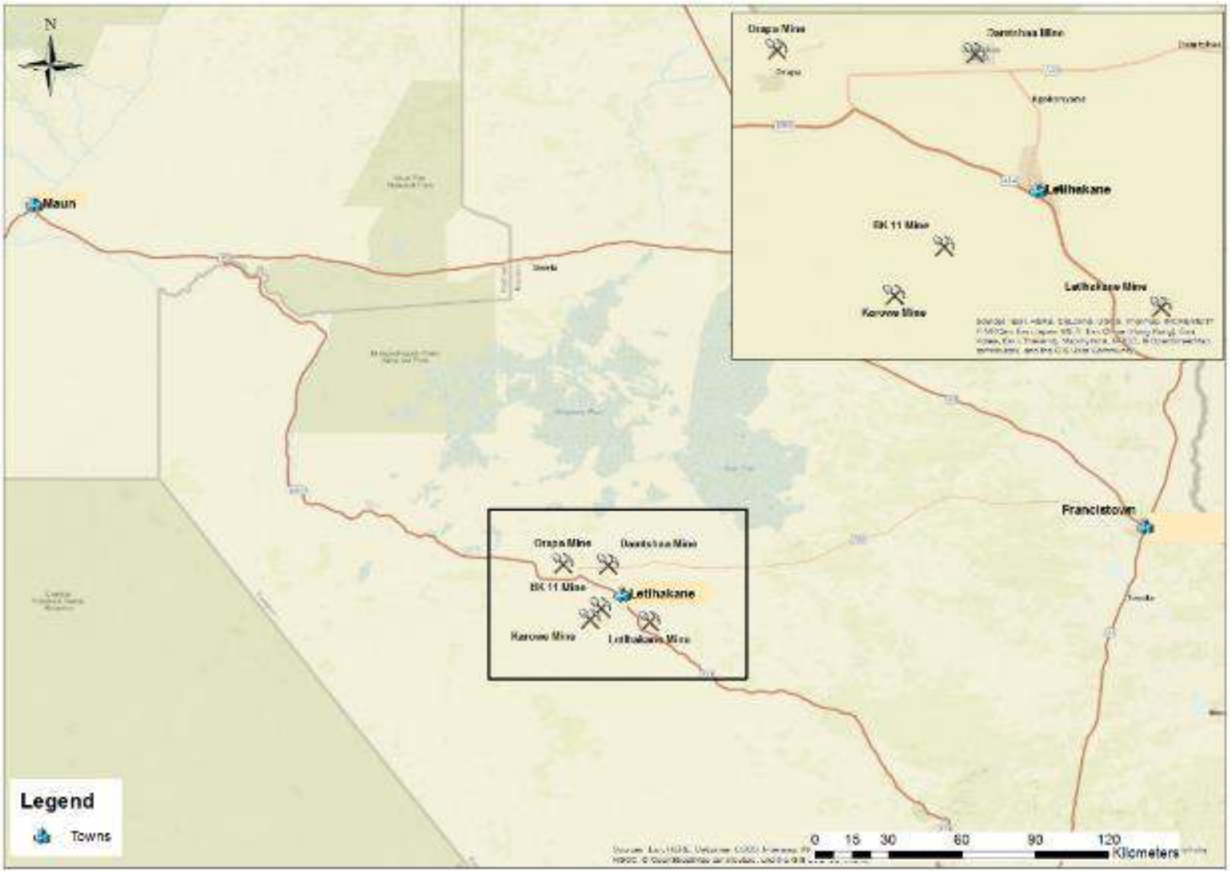


Figure 23.1 – Figure showing Adjacent Properties to Karowe Mine

23.1 Orapa Mine

Orapa is the second largest commercially exploited kimberlite in the world. Table 23.1 shows a summary of Orapa mine operations.

Table 23.1 – Summary of Orapa Mine

Owner	Debswana Diamond Mining Company (Pty) Ltd
Mining Licence[1]	Valid up to 2029
Mining Started[1]	1971
Mining Method[1]	Open Pit
Grade[1]	101.3 cpht (Measured and Indicated)
Geology[1]	Kimberlite AK/1
Life of Mine[1]	14 Years up to 2030
Resource/Reserves[1]	295.4 Mt (Measured and Indicated)

[1]Source: Anglo American Ore Reserves and Mineral Resources Report 2016

23.2 Letlhakane Mine

The Letlhakane Mine produces diamonds of very high quality. There are plans to re-treat tailings at the Letlhakane Mine.

Table 23.2 shows a summary of Letlhakane mine.

Table 23.2 – Summary of Letlhakane Mine

Owner	Debswana Diamond Mining Company (Pty) Ltd
Mining Licence[1]	Up to 2029
Mining Started[1]	1977
Mining Method[1]	Open Pit
Grade[1]	31.7 cpht (Measured and Indicated)
Geology[1]	Kimberlite DK/1 and DK/2
Life of Mine[1]	1 Year up to 2017
Resource/Reserves[1]	22.2 Mt (Measured and Indicated)

[1]Source: Anglo American Ore Reserves and Mineral Resources Report 2016

23.3 Damtshaa Mine

The Damtshaa Mine is designed to exploit four relatively low-grade kimberlites which were discovered by De Beers in the 1960s and 1970s. Table 23.3 shows a summary of Damtshaa mine.

Table 23.3 – Summary of Damtshaa Mine

Owner	Damtshaa Mine
Mining Licence[1]	Up to 2029
Mining Started[1]	2002
Mining Method[1]	Open Pit
Grade[1]	25.0 cpht (Measured and Indicated)
Geology[1]	BK/9 and BK/12
Life of Mine[1]	18 Years up to 2034
Resource/Reserves[1]	4.4 Mt (Measured and Indicated), 19 Mt (Inferred)

[1]Source: Anglo American Ore Reserves and Mineral Resources Report 2016

23.4 Firestone Diamonds BK11

Firestone Diamonds plc, listed on the London Alternative Investment Market (“AIM”) is the controlling partner and operator in Monak Ventures (Pty) Ltd (Firestone Diamonds plc 90%; other interests 10%) which has developed the BK11 kimberlite, 5.2 km northeast of the Karowe Mine. The mine has been on care and maintenance since 2012[1].

Following a decision to focus on its flagship Liqhobong Diamond Mine in Lesotho, Firestone commenced a disposal process for all its assets in Botswana. In May 2017 Firestone entered into a conditional option agreement for the potential disposal of its Botswana operations, which include its interest in the BK11 mine, to Amulet Diamond Corporation for a total potential consideration of US$5.1 million in cash. Table 23.4

Table 23.4 – Summary of BK11

Owner	Firestone Diamonds
Mining Licence	Unknown
Mining Started[1]	2010
Mining Method[1]	Open Pit
Grade[1]	8.5 cpht
Geology[1]	Kimberlite BK11
Life of Mine[1]	10 Years
Resource/Reserves[1]	11.5 Mt

[1]*Firestone website*

24 Other Relevant Data and Information

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

25.1 Mineral Resource Estimate

A Mineral Resource estimate was prepared by Mineral Services Canada Ltd (MSC) and published by Lucara on 18th December 2013 (Lynn et al, 2014). No changes have been made to the geological model, bulk density or tonnage estimates stated in 2013, indicated resources have been adjusted for open pit depletion. The grade estimates made in 2013 were updated in 2016. The value models of 2013 have been updated in this report based on the substantially larger diamond sales dataset now available following an additional 4 years of active production. These updates to the grade and value estimates have been incorporated into the 2013 block model in which the original resource estimate was hosted, forming the basis for this PEA. Details of these revisions, and the basis for all estimates, are provided in Section 14. The Mineral Resource statement is presented in Table 1.2.

25.2 Conclusions

The Mineral Resource estimate and underground mining schedule produced for this PEA study have been used as the basis for the financial model for the project. The financial model indicates that the underground project has positive economics up to its scheduled close in 2036. At current economics, the resultant NPV is US$451 Million at a 5% discount rate, and a free cash flow of US$820 Million after taxes.

Table 25.1 shows a summary of the underground project economic sensitivities and key operational parameters.

Table 25.1 – Economic Sensitivities and Key Operational Parameters

Parameter	Unit	Base Case
Rough diamond price – South lobe (2017)	US$/carat	$730
Rough Diamond Annual Real Diamond Price Escalator	%	2.5
After-Tax Undiscounted Net Cash Flow	US$M	$820
After-Tax NPV (5%)	US$M	$451
After-Tax NPV (8%)	US$M	$318
After-Tax IRR	%	38.9%
Pre-Tax Undiscounted Net Cash Flow	US$M	$901
Payback Period (pre-tax)	years	2.5

Production	Average Annual
Rough Diamonds million (carats)	2.72

Operating Costs	US$ per tonne treated
Kimberlite (US$/t treated)	$49.4
Diamonds (US$/carat recovered)	$407.70
All-In Sustaining Costs[1]	US$ per tonne treated
Kimberlite (US$/t treated)	$54.18
Diamonds (US$/carat recovered)	$411.72

Production Costs	$/t
Operating Cost	$49.36
Overhead Costs	$8.93

It is the conclusion of the QP's (refer to Section 2.3) that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QP's are not aware of any fatal flaws for the underground project.

The PEA is preliminary in nature, in that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable the mining target to be categorized as Mineral Reserves.

Using the assumptions highlighted in this report, the Karowe underground project offers sufficient economic potential to warrant the project to be advanced to the next stage of study (Pre-Feasibility Study).

25.3 Risks

The main risks identified, by means of a workshop held at RHDHV including key members of the mine, owner's team and the RHDHV team, with the Karowe underground project are shown in Table 25.2 below.

Table 25.2 – Summary of Main Project Risks

Risk	Explanation/Potential Impact	Risk Mitigation
Mineral Resource Estimate	Conversion of Inferred to Indicated categories, additional diamond sampling required, delay to PFS and FS studies, PEA outcome not realised.	Resource update underway with additional drilling, geological interpretation, current Indicated and Reserve resources de-risked through mining and diamond sales since 2012.
Geohydrology	Insufficient understanding of the sources, flow paths and quantities of ground water inflow from hydrogeological units and regional aquifers below the open pit.	Increased definition of the geological units for geotechnical and hydrogeological predictions. Detailed testing of the (discrete) units below the Mosolotsane. Update structural interpretation. Revise conceptual hydrogeological model and update water balance. Update numerical model.
Geotechnical	Insufficient Geotechnical descriptive detail available to allow for accurate geotechnical sub zoning within the major geological units for the underground mine.	Additional geotechnical drilling and logging is required. Logging to focus on definition of sub zones within the geological units and the complete stratigraphic column to be accessed via open pit and the proposed underground mine.
Mining	Mine designs and associated work were prepared at a conceptual level, which will require additional engineering and design, to ensure that the level of design is commensurate with any financial decisions that Lucara is considering related to the underground mine.	More detailed mining trade off studies are required to select the appropriate access options and optimised mining method. Detailed planning based on input from the proposed Geohydrology and Geotechnical programmes is required to further define the mine design criteria to the appropriate level.
Capital and Operating Cost	The ability to achieve the estimated CAPEX and OPEX costs are	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost reduction measures

Risk	Explanation/Potential Impact	Risk Mitigation
	important elements of project success.	would assist in the support of reasonable cost estimates.
Development Schedule	Development and production scheduling has not been performed in detail. Experience factors were applied to assist in estimating reasonable completion dates. Detailed scheduling could result in delays to the initial production dates and to the production ramp up schedules.	Additional data required for the assumptions made in this report are listed below: Geotechnical data at depth to ensure the mining method is suitable and ground support requirements can be designed to suit. Evaluation of hydrological data to confirm that water will not significantly impact costs and schedules beyond that which has been assumed.
Human Resources	Sourcing and training of a large workforce presents challenges, particularly moving from an open pit to underground mining. Mining productivities and costs would be affected adversely if sufficient numbers of trained workers cannot be provided.	The early search for professionals, as well as maintaining competitive salaries, flexible work schedules and benefits all help to identify, attract and retain critical people.

25.4 Opportunities

No	Opportunity
1	During the PFS study, consideration will be given to other mining methods to reduce operating costs, dilution and tramp steel reporting to the plant.
2	To plan an early access to the proposed underground workings in order to facilitate dewatering and de-risk the underground access and production ramp up.
3	Underground mining optimization opportunities exist at higher throughputs which will have a positive impact on operating costs. Increased mined kimberlite delivery and plant upgrade requirements will be investigated.
4	Rim tunnel spacing is currently set at 25m vertical spacing. Should the geotechnical data show improved stability, the distance between the rim tunnels may be increased to up to 30m, reducing ongoing development costs.
5	The PEA assumes purchasing all mobile equipment. It is possible that this equipment may be leased resulting in lowered costs.
6	It may be possible to decrease the mine operating costs through the use of automated production load-haul-dump equipment.

26 Recommendations

Karowe mine as an established operation, has an existing process plant, engineering infrastructure and an established sales and marketing network, which significantly reduces the risks associated with establishing a new mine. It is RHDHV's opinion that the financial analysis of this PEA of the Karowe underground project has positive economics and warrants consideration for advancement to a Pre-feasibility study (PFS).

26.1 Legal and Tenure

The underground mine life has the potential to extend to 2037. The mining licence for Karowe mine will need to be extended before 2023.

26.2 Resource

An update to the 2013 resource model is currently in progress to convert inferred mineral resources to the indicated category at the required depths to support an underground mine to 400 mamsl. Conversion of inferred resources below 400 mamsl would require additional drilling and grade determination. Revenue models, based on value models may be updated utilising actual diamond sales data going forward. Drilling to determine the depth extent of the centre and north lobes to support potential underground mining is not covered by the current PEA.

26.3 Geotechnical

A geotechnical drilling programme in the surrounding host rock and kimberlite is proposed. The programme requires significant additional geotechnical drilling and information gathering to take place during 2018, in order to obtain the appropriate density of information as required for a future feasibility level study.

26.4 Geohydrology

The recommendations to be addressed as part of the PFS phase includes the following:

- Further investigation of the sub-lithologies of the country rock in order to understand the impacts on mining and engineering design.
- Update of the structural and country rock model.
- Detailed mine water balance.
- Integration of the hydrogeology and geotechnical design with the open pit and underground mine planning and engineering designs.

26.5 Mining

It is recommended that the following work to be undertaken for the PFS:

- Trade off studies for the underground mine should be undertaken for various mining methods, automation, access and ground handling options.
- More detailed development and production schedules should be prepared to obtain better estimates of pre-production period and the time required to ramp up to full underground production. This would inform the timeline requirements of the current open pit mining to appropriately transition from open pit to underground operations.
- Detailed sequencing of the access development and pre-production activities.
- Capital and operating cost estimates to be further refined.
- To plan an early access to the proposed underground workings in order to facilitate dewatering and de-risk the underground access and production ramp up.
- More detailed ventilation studies need to be conducted as part of the PFS.

26.6 Processing and Tailings Storage Facility

For the PFS study, it is recommended that a Geometallurgical data base be established in order to record any pertinent data or information such as ore hardness, density and particle size distribution that is gathered from the current operations and exploration/drilling programmes. This database would then be used to provide the expected underground ROM feed characteristics envelope as inputs for modelling the process.

The potential for tramp metal to report with the plant feed must also be assessed once the mining method has been chosen in order to cater for additional detection and removal as necessary, and assess the impact on the existing process plant.

For the tailings storage facility design, a more detailed assessment will be required to inform the design requirements for the PFS. This will include but not be limited to the following:

- A Digital Terrain Model (DTM) survey of the current area available and the additional area required for the underground expansion.
- Foundation geotechnical investigation on the new footprint.

26.7 Infrastructure

The following will need to be undertaken as part of the PFS:

- Review the power requirements to support the mine's activities including the underground operations for both construction as well as steady state production.
- A revised water balance including the transition from open pit to underground mining. Potable water requirements for the underground will also need to be assessed.
- The current surface infrastructure should be reviewed in line with the underground requirements and would typically include an assessment to support the open pit to underground mining transition, steady state underground production and processing operations.
- Planning for construction activities for the underground requirements.

26.8 Environmental, Social and Community

For the PFS, a more detailed assessment will be required to determine the potential need for any additional land. Assuming no additional land is required for the proposed underground operation, the existing Environmental Impact Assessment (EIA) will require updating as part of Lucara's regulator and stakeholder engagement process. This would include an update of the mine closure plan.

Should additional land be required for the underground operation, the Project Mining Lease and its EIA would require public scoping and review. Following Lucara's regulator and stakeholder engagement process, the mine would be required to disclose the objectives of the underground operation as well as any new impacts that may be associated with the project implementation.

More detailed investigations in terms of the EIA update would include the development of a formal, quantitative and qualitative surface and groundwater model for the project, a recalculation of the mine closure liability, and targeted investigations into new opportunities for socio-economic project and commercial business development which would result in broader socio-economic benefits and thereby strengthen the project's social license to operate.

26.9 Costs

The estimated order of magnitude costs for the undertaking of a Pre-feasibility study is shown in Table 26.1 below.

Table 26.1 – Summarised Pre-Feasibility Costs

Recommended Work for PFS	Cost Estimate (US$)
Drilling	22.7 Million
Pre-feasibility Study	1.2 Million
Total	24 Million

Royal HaskoningDHV

Lucara Diamond Corp.

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Appendix 1 – Glossary of Terms

Glossary

Term	Meaning
amsl	Above Mean Sea Level
Archaean	The geologic eon from about 3,800 to 2,500 million years ago
basalt	A mafic volcanic rock composed chiefly of plagioclase and pyroxene. It is the extrusive equivalent of gabbro.
BTC	Botswana Telecommunications Corporation
carbonate	A rock, usually of sedimentary origin, composed primarily of calcium, magnesium or iron and CO3. Essential component of limestones and marbles.
chromite	An iron chromium oxide: FeCr2O4. It is an oxide mineral belonging to the spinel group.
density	A measure of the amount of matter contained by a given volume.
dip	The angle at which a planar feature, such as bedding or schistosity, is inclined from the horizontal.
dolerite	Medium-grained intrusive igneous rock of basaltic composition (plagioclase + clinopyroxene)
DTM	Digital terrain Model
dunite	An igneous, plutonic rock, of ultramafic composition, with coarse-grained texture. The mineral assemblage is greater than 90% olivine.
dyke	A tabular body, typically of igneous rock, which cuts across the structure of another older rock.
fault	A crack in the earth's crust resulting from the displacement of one side with respect to the other.
fancy	A term used to describe diamonds of unusual and rare colour, for example blues, pinks and greens
gabbro	A mafic igneous rock composed chiefly of plagioclase and clinopyroxene, sometimes with olivine. It is the intrusive equivalent of basalt.
gneiss	A common and widely-distributed metamorphic rock having bands or veins, but not schistose.
grade	A measure that describes the concentration of a valuable natural material in a mineral deposit.

granite	A common widely occurring type of intrusive, felsic, igneous rock. Granite usually has a medium- to coarse grained texture.
igneous	Pertaining to a rock that has crystallized out of a melt.
intrusion	Liquid rock that forms under Earth's surface. Magma from under the surface is slowly pushed up from deep within the earth into any cracks or spaces it can find, sometimes pushing existing country rock out of the way, a process that can take millions of years. As the rock slowly cools into a solid, the different parts of the magma crystallize into minerals
kimberlite	A type of potassic, carbon dioxide containing, volcanic rock (peridotite) best known for sometimes containing diamonds.
lithology	A description of a rock's physical characteristics visible at outcrop, in hand or core samples or with low magnification microscopy, such as colour, texture, grain size, or composition.
LHD	Load Haul Dumper
mafic	A type of rock that is rich in magnesium and iron magnetite An iron oxide mineral with the chemical formula Fe_3O_4 and a member of the spinel group
mamsl	Metres above mean sea level
mbc	Meters below collar
mbs	Meters below surface
MD	Maximum Demand
mica	A group of hydrous aluminosilicate minerals characterized by highly perfect cleavage, so that they readily separate into very thin leaves.
NMD	Notified Maximum Demand
Natural Remanant Magnetisation (NRM)	The permanent magnetism of a rock caused by the alignment of magnetic particles in the rock with the Earth's magnetic field at the time the rock formed.
olivine	A magnesium iron silicate mineral with the formula $(Mg,Fe)_2SiO_4$.
PEA	Preliminary economic assessment
PFS	Pre-feasibility study
peridotite	A dense, coarse-grained igneous rock, consisting mostly of the minerals olivine and pyroxene. Peridotite is ultramafic, as the rock contains less than 45% silica. It is high in magnesium, reflecting the high proportions of magnesium-rich olivine, with appreciable iron.

PLI	Point Load Index
QAQC	Quality assurance and quality control.
quartz	The most abundant mineral on the earth's surface, of chemical composition silicon dioxide, SiO_2.
RMR	Rock Mass Rating
RPI	Residual Passive Inflow
SOPs	Standard Operating Procedures
SG	Specific gravity. The ratio of the density (mass of a unit volume) of a substance to the density (mass of the same unit volume) of a reference substance.
shale	A fine-grained, clastic sedimentary rock composed of mud that is a mix of flakes of clay minerals and tiny fragments (silt-sized particles) of other minerals, especially quartz and calcite.
special	Refers to a gem quality diamond of greater than 10.8 ct. Type IIa diamond Diamonds characterised by their very low (<20 ppm) nitrogen contents. The Type IIa stones often have top quality white colours (D-G), a consequence of their low nitrogen contents. They include the largest gem diamond ever found, the 3,106 ct Cullinan, recovered from the Premier Mine, South Africa, as well as gems like the legendary Koh-i-noor, from India.
tpd	Tonnes per day
ultramafic rock	(also referred to as ultrabasic) rocks are igneous and meta-igneous rocks with very low silica content (less than 45%), generally >18% MgO, high FeO, low potassium, and are composed of usually greater than 90% mafic minerals (dark coloured, high magnesium and iron content).
weathering	Mechanical or chemical breaking down of rocks in situ by weather or other causes.
ZOR	Zone of Relaxation

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Appendix 2 – Certificates of Qualified Persons

Mr Guilluame Johannes (Johan) Oberholzer

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CERTIFICATE OF QUALIFIED PERSON

I, Guilluame J Oberholzer with business address at 21 woodlands Drive, Woodmead, Johannesburg, South Africa do hereby certify that:

- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 1.1, 1.2, 1.5, 1.10, 1.11, 1.12, 2, 3, 4, 5, 15, 18.1, 18.2, 21, 22, 23, 24, 25.2, 25.3, 25.4, 26.1, 26.7, 26.9 and 27 of the report.
- I have personally visited the Karowe Mine near Letlhakane in Botswana from 9 to 11 September 2017 as part of the due diligence necessary to compile this report.
- I hold a BSc Degree in Electrical Engineering, University of Pretoria, and am a registered Professional engineer with the Engineering Council of South Africa (ECSA). I am also a current member of the following processional bodies:
    - South African Institute of Electrical Engineers (Senior Member)
- I have practiced my profession as an electrical engineer within the international mining industry continuously for 32 years.
- I am currently the Manager Engineering (Mining) at Royal HaskoningDHV, an Independent Engineering Consultancy.
- I have not received, nor do I expect to receive, any interest, directly or indirectly, from the Issuer, or of any affiliate thereof, and I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the public filing of the technical report titled " NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017 (the "Technical Report") by Lucara Diamond Corp. (the issuer). I also consent to any extracts from or a summary of the Technical Report being included in public disclosures by the issuer.

Dated this 27th November 2017.

Mr GJ Oberholzer (signed and sealed)
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Norman George Carroll Blackham

Blackham Consulting

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CERTIFICATE OF QUALIFIED PERSON

I, Norman George Carroll Blackham with business address at 27 Boskruin Manor, Bateleur Place, Bromhof, Johannesburg, 2188, do hereby certify that:

- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 1.9, 17, 18.3, 25.3, 25.4, 26.6 and 27 of the report.
- I have personally visited the Karowe Mine near Letlhakane in Botswana from 9 to 11 September 2017 as part of the due diligence necessary to compile this report.
- I hold a Bachelor of Science (Honours) degree from the Royal School of Mines, Imperial College, London.
- I am an elected Fellow of the Southern African Institute of Mining and Metallurgy in good standing (FSAIMM)
- I have practiced my profession as a mineral processing engineer within the mining industry continuously for 41 years.
- I am currently owner of and Principal Consultant with Blackham Consulting, a provider of mineral processing optimisation, business improvement, security auditing and operational consulting services to the mining industry.
- I have not received, nor do I expect to receive, any interest, directly or indirectly, from the Issuer, or of any affiliate thereof, and I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have not had prior involvement with the property that is the subject of this Independent Technical Report.
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the public filing of the technical report titled "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017 (the "Technical Report") by Lucara Diamond Corp. (the issuer). I also consent to any extracts from or a summary of the Technical Report being included in public disclosures by the issuer.

Dated this 27th November, 2017.

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CERTIFICATE OF QUALIFIED PERSON

I, John Anthony Cox with business address at Building No 5, Country Club Estate, 22 Woodlands Drive, Woodhead, Gauteng, South Africa, 2191, do hereby certify that:

- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 1.8, 16.3, 16.4, 16.5, 25.3, 25.4, 26.5 and 27 of the report.
- I have not personally visited the Karowe Mine near Letlhakane in Botswana as part of the due diligence necessary to compile this report.
- I hold a Bachelor of Science (Engineering) degree in Mining from the University of London in 1961.
- I am a Fellow of the South African Institute of Mining and Metallurgy and a Professional Engineer of the Engineering Council of South Africa (ECSA)
- I have worked as a mining engineer for a total of 55 years in a number of roles, including mining operations, mining projects and mining consulting. I have conducted mining reviews for a wide range of commodities, mining methods and styles of mineralisation.
- I currently have no ties to the Issuer and am totally independent of the issuer within the meaning of Section 1.5 of NI 43-101.
- I have had consulting involvement with several projects targeting massive diamondiferous orebodies and making use of the same / similar mining method(s). I have also had operational involvement with massive base mineral orebodies using the same mining methods
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the public filing of the technical report titled " NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017 (the "Technical Report") by Lucara Diamond Corp. (the issuer). I also consent to any extracts from or a summary of the Technical Report being included in public disclosures by the issuer.

Dated this 27th November 2017.

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CERTIFICATE OF QUALIFIED PERSON

I, Jody Thompson with business address at 15 Kingfisher Place, Bromhof, Randburg, 2154 do hereby certify that:

- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 1.6, 16.1, 25.3, 25.4, 26.3 and 27 of the report.
- I have personally visited the Karowe Mine near Letlhakane in Botswana on 9 to 11 September 2017 as part of the due diligence necessary to compile this report.
- I hold a Bachelor of Engineering degree in Mining from the University of Tshwane and am the holder of the South African Chamber of Mines' Rock Engineering Certificate (COMREC No 399). I am also a past / current member of the following processional bodies
    - South African National Institute of Rock Engineers SANIRE,
    - International Society of Rock Mechanics ISRM, and
    - South African Institute of Mining and Metallurgy SAIMM.
- I have practiced my profession as a mine geotechnical engineer within the mining industry continuously for 17 years.
- I am currently practicing as an Associate Geotechnical Consultant to Royal HaskoningDHV, an Independent Engineering Consultancy. I also service several other clients through my company TREM Engineering cc.
- I have not received, nor do I expect to receive, any interest, directly or indirectly, from the Issuer, or of any affiliate thereof, and I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had both consulting and operational involvement with several projects targeting massive diamondiferous orebodies and making use of the same / similar mining method(s).
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the public filing of the technical report titled " NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017 (the "Technical Report") by Lucara Diamond Corp. (the issuer). I also consent to any extracts from or a summary of the Technical Report being included in public disclosures by the issuer.

Dated this 27th November 2017.

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I, Kym L Morton with business address at 22-24 Central Rd LANSERIA 1748 South Africa do hereby certify that:

- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 1.7, 16.2, 25.3, 25.4, 26.3 and 27 of the report.
- I have personally visited the Karowe Mine near Letlhakane in Botswana numerous times as part of the due diligence necessary to compile this report.
- I hold a BSc Honours, Kings College London University, MSc Hydrogeology University College London, PhD Imperial College London, MBA Imperial College London and am a Registered Chartered Geologist in the UK. I am also a past / current member of the following processional bodies
    - South African National Association of Natural Scientists (Pr Sci Nat)
    - Fellow of the Geological Society UK (FGS)
    - Member of the Geological Society South Africa (MGSSA)
    - Fellow of the South African Institute of Mining and Metallurgy FSAIMM.
- I have practiced my profession as a mining hydrologist and Hydrogeologist/Geohydrologist within the international mining industry continuously for 36 years.
- I am currently practicing as an Associate Geohydrologist Consultant to Royal Haskoning DHV, an Independent Engineering Consultancy. I also service several other clients through my company KLM Consulting Services Pty Ltd
- I have not received, nor do I expect to receive, any interest, directly or indirectly, from the Issuer, or of any affiliate thereof, and I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had both consulting and operational involvement with several projects targeting massive diamondiferous orebodies and making use of the same / similar mining method(s).
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
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Dated this 27th November 2017.

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- I have contributed to the report titled, "NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017, for Lucara Diamond Corp. (the "Issuer"),
- and I am responsible for Sections 20, 26.8 and 27 of the report.
- I have personally visited the Karowe Mine near Letlhakane in Botswana as part of the due diligence necessary to compile this report.
- I hold a BA (Hons.) and an M.A. from Queen's University Kingston, Ontario, Canada and a PhD in Restoration Ecology/ Rehabilitation from the University of the Witwatersrand, Johannesburg, South Africa.
- I have practiced the profession of Corporate Environmental Manager, Environmental, Social & Governance (ESG) analyst, and Environmental Consultant within the international mining industry continuously for 23 years.
- I am currently practicing as an Associate Environmental Consultant to Royal Haskoning DHV, an Independent Engineering Consultancy. I also service several other clients through my company Reichardt & Reichardt (Partnership)
- I have not received, nor do I expect to receive, any interest, directly or indirectly, from the Issuer, or of any affiliate thereof, and I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.
- I have had both consulting and operational involvement with several projects targeting diamondiferous, metalliferous, carboniferous and other orebodies and making use of the same / similar mining method(s).
- I have read National Instrument 43-101 and Form 43-101F1 and, by reason of education and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101. The parts of the technical report for which I am responsible have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the public filing of the technical report titled " NI43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" dated 27th November 2017 (the "Technical Report") by Lucara Diamond Corp. (the issuer). I also consent to any extracts from or a summary of the Technical Report being included in public disclosures by the issuer.

Dated this 27th November 2017.

Dr M Reichardt (signed and sealed)
Reichardt & Reichardt

CERTIFICATE OF AUTHOR

I, Tom Nowicki, P.Geo., do hereby certify that:

- I am currently employed as a Senior Principal Geoscientist with Mineral Services Canada Inc. with an office at 501 – 88 Lonsdale Avenue, North Vancouver, BC, V7M 2E6, Canada.

- This certificate applies to the technical report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project", with an effective date of 27 November 2017, (the "Technical Report") prepared for Lucara Diamond Corporation ("the Issuer").

- I am a Professional Geoscientist (P.Geo. #30747) registered with the Association of Professional Engineers, Geologists of British Columbia.

  - I am a graduate of the University of Cape Town having obtained the degree of Bachelor of Science (Honours) in Geology in 1986 and a Ph.D. Degree in geochemistry in 1998. I am a graduate of Rhodes University (Grahamstown, South Africa) having obtained the degree of Masters of Science in Economic Geology in 1990. I have been employed as a full-time geoscientist in the mineral exploration and mining fields in 1987 and 1988, from 1990 to 1993 and from 1998 to present.

  - I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

- I visited the Karowe Diamond Mine on the 3rd and 4th July 2013.

- I am responsible for Sections 12.1 to 12.6 and Section 14 (other than Section 14.5) of the Technical Report.

- I am independent of the Issuer and related companies as independence is described in Section 1.5 of NI 43-101.

- I was previously involved with the Karowe Diamond Mine as the independent QP responsible for preparation of the 2013 Mineral Resource Estimate for Karowe, as documented in "Karowe Diamond Mine, Botswana, NI 43-101 Independent Technical Report, Effective Date 31 December 2013".

- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 27th November 2017
Signing Date: 27th November 2017

(original signed and sealed) "Tom Nowicki, P.Geo.."

Tom Nowicki, P.Geo.

Dr. John Armstrong

CERTIFICATE OF QUALIFIED PERSON

I, John Armstrong PhD, P.Geol. (NT/NU) of Vancouver, British Columbia, Canada do hereby certify that:

- I am Vice President, Mineral Resources, for Lucara Diamond Corporation (2000-885 West Georgia Street, Vancouver, British Columbia, V6C 3E8, Canada) and a co-author of the "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project" (the Technical Report), dated 27 November 2017, and prepared for Lucara Diamond Corporation.
- I graduated from the University of Western Ontario in 1989 (H.BSc.Geology), and the University of Western Ontario in 1997 (Doctor of Philosophy (PhD), and have practiced my profession continuously since graduation.
- I am a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License # 1697).
- I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, past relevant work experience and affiliation with a professional association, I fulfil the requirements of a "qualified person" for the purposes of NI 43-101.
- I have visited the Karowe Diamond Mine on a regular basis since October 2013 with the most recent visit being October 2017.
- I am responsible for Sections 1.3, 6, 7, 8, 9, 10, 12.7, 14.5, 19, 25.1, 25.3, 25.4, 26.2,27, of the Technical Report
- I am not independent of Lucara Diamond Corporation due to my position as an Officer of the Corporation, as defined in section 1.5 of NI 43-101.
- I have been involved with mining, production, and diamond sales activities at the Karowe Diamond Mine since October 2013.
- I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with such instrument and form.
- As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 27th November 2017 at Vancouver.

(signed) "John Armstrong"

John Armstrong, P.Geol. (NT/NU)
Vice President, Mineral Resources
Lucara Diamond Corporation