2018 Technical Report for the Karowe Mine: Updated Mineral Resource Estimate

Report prepared for:

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Mineral Services
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Lucara Diamond
Corp.

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LIST OF ABBREVIATIONS

Units of measurement	
Symbol	**Description**
$/ct	United States dollars per carat
$/ct	United States dollars per carat
% ct	Percentage carat
cm	Centimetre
cpht	Carats per hundred tonne
cpm^3	Carats per cubic metre
cps	Carats per stone
cpt	Carats per tonne
ct	Carat
ha	Hectare
kg	Kilogram
km	Kilometre
km^2	Square kilometer
kt	Thousand metric tonne
kV	Thousand volts
l	Litre
m	Metre
m/yr	Metres per year
m^3	Cubic metre
m^3/hr	Cubic metres per hour
m^3/yr	Cubic metres per year
masl	Metres above mean sea level
mbs	Metres below surface
Mct	Million carat
mm	Millimetre
Mm^3	Million cubic metre
Mt	Million metric tonne
Mtpa	Million tonnes per annum
^oC	Degrees Celsius
spkg	Stones per kilogram
st	Stone
tpm^3	Tonnes per cubic metre
USDm	Million United States dollars

Other abbreviations					
Symbol	**Description**	**Symbol**	**Description**	**Symbol**	**Description**
3D	Three dimensions	IJK	Datamine identifier for block cell locations	ROM	Run of mine
ADT	Articulated dump truck	KDM	Karowe Diamond Mine	RST	Regular stone tender
BCM	Bank cubic metre	l	Litre	RTK	Real-time kinetic
BCOS	Bottom cut-off screen	LDD	Large-diameter drill(ing)	RVK	Resedimented volcaniclastic kimberlite
BD	Bulk density	LDV	Light duty vehicle	S	South
BPC	Botswana Power Corporation	LG	Low grade	SC	Standard Chartered
BTC	Botswana Telecomms Corp	LOM	Life of mine	SFD	Size frequency distribution
BWP	Botswana Pula (currency)	M	Million, suffix or prefix to other abbreviations	SP	Stockpile
BWPm	Million Botswana Pula	MCF	Mine call factor	Sph	Spherical
CIM	Canadian Institute of Mining and Metallurgy	MCRP	Mine Closure and Rehabilitation Plan	SRC	Saskatchewan Research Council
CSR	Community social responsibility	MD	Maximum demand	SRK	SKR Consulting
DGPS	Differential global positioning system	MG	Medium grade	TK	Tuffisitic kimberlite
DMS	Dense media separation	MK	Magmatic/coherent kimberlite	TLB	Tractor loader backhoe
DTC	Diamond Trading Company	ML	Mining License	UCS	Unconfined compressive strength
DTM	Digital terrain model	MVA	Megavolt amps	UG	Underground
DXF	Autocad extension file	MVK	Massive volcaniclastic kimberlite	UTM	Universal Transverse Mercator
E	East	N	North	VK	Volcaniclastic kimberlite
EIA	Environmental Impact Assessment	NI 43-101	Canadian National Instrument 43-101	VLG	Very low grade
EMP	Environmental Management Plan	NMD	Notified maximum demand	W	West
ESG	Environmental, Social and Governance	NPV	Net present value	WGS84	World Geodetic System 84
EST	Exception stone tender	OKF	Orapa kimberlite field	X	Easting direction
Expo	Exponential	OPU	Overall plant utilization	XRT	X-Ray transmission
FALC	Fort al la Corne	OP	Open pit	Y	Northing direction
FEL	Front end loader	OSA	Overall slope angles	Z	Elevation (masl)
FOV	Field of view	PK	Pyroclastic kimberlite	Z	Riemann zeta graphical function (SFD plots)
GDV	Government diamond valuator	PL	Prospecting license		
GEMS	Dassault Systemes Geovia GEMS™	PPL	Plain polarized light		
GNSS	Global Navigation Satellite System	PTK	Pyroclastic tuffisitic kimberlite		
GR	Grainer	Q	Year quarter (with number suffix)		
GT	Geotechnical	QA/QC	Quality control / quality assurance		
HG	High grade	QP	Qualified Person		
HK	Hypabyssal kimberlite	RCPT	Recoverable carats per tonne		
HY1	First half of year	RF	Revenue factor		

1. Executive summary

1.1 Introduction

The Karowe Mine is an existing open pit diamond mine extracting and processing ore from the AK6 kimberlite in the Central District of Botswana. The Karowe Mine has been in production since April 2012, operated by Boteti Mining (Pty) Ltd. (Boteti), a wholly owned subsidiary of Lucara Diamond Corp (Lucara).

Mineral Services Canada (MSC) has been retained by Lucara to integrate results from recent (2017) evaluation work with previous evaluation datasets and update the Mineral Resource Estimate for AK6. This Independent Technical Report has been compiled by MSC on behalf of Lucara to fulfil reporting requirements for public disclosure of Mineral Resources as outlined by Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. The updated Mineral Resource Estimate has been used to re-state the Mineral Reserve Estimate for the Open Pit mining activity at Karowe. In addition, the updated Mineral Resource Estimate will be used to support an ongoing Feasibility Study on an open pit to underground mining transition and ultimately an underground mining operation at Karowe.

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 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 its 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 ~1,020 m above sea level and is covered by sand savannah with 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 ML 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 national grid on commercial terms. Water for the mine is derived from a strong aquifer.

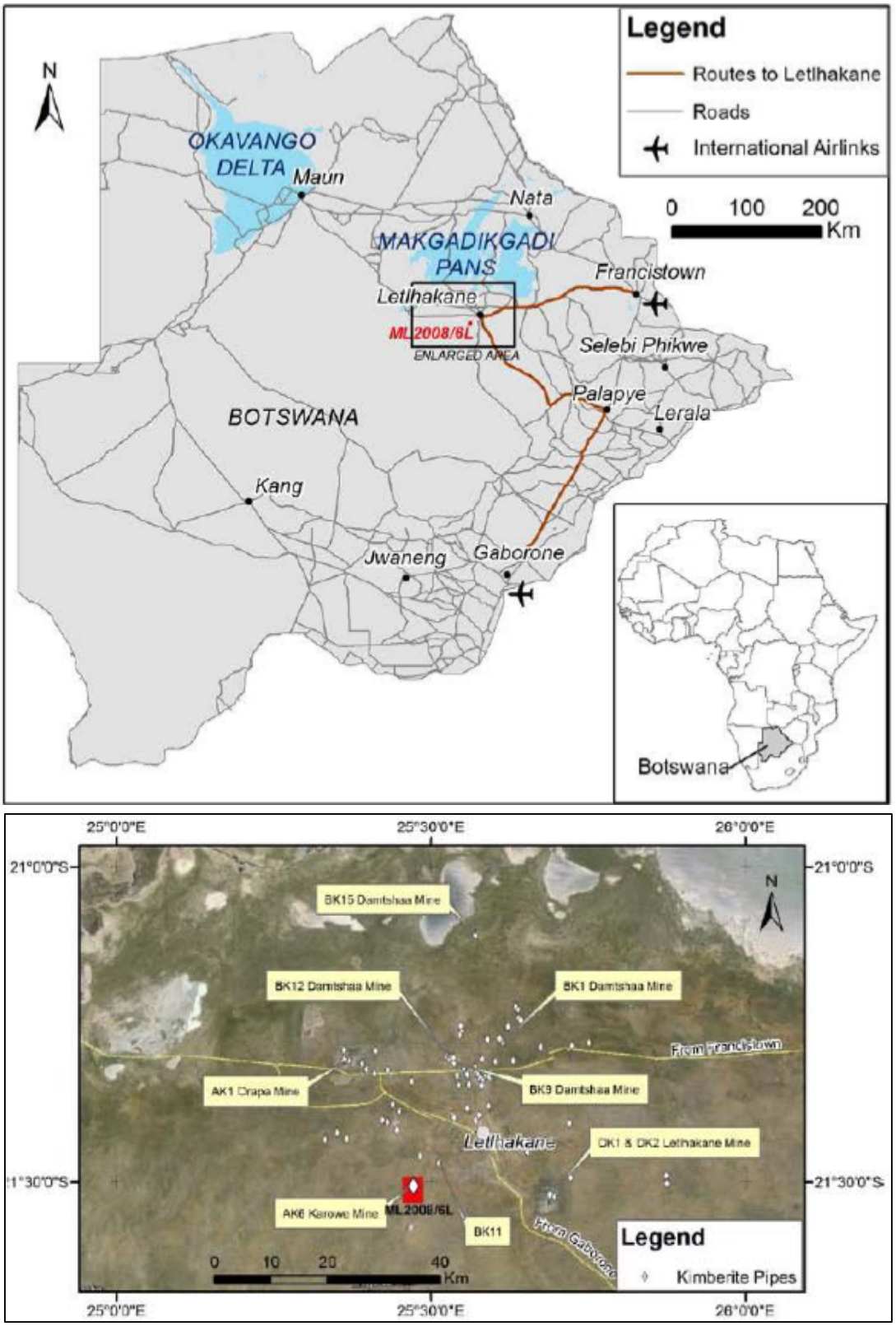

Figure 1-1: Locality map of the Karowe Mine and adjacent mines in Botswana.

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

The OKF includes at least 83 kimberlite bodies of post-Karoo age. Five of these (AK1, BK9, DK1, DK2 and AK6) 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 (North, Centre and South Lobes) that taper with depth into discrete roots.

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 North and Centre Lobes exhibit significant textural complexity (reflected in apparent variations in degree of fragmentation and proportions of country rock xenoliths) whereas the South Lobe is more massive and internally homogeneous.

Kimberlite material has been grouped into mappable units (Table 1-1) based on geological characteristics and interpreted grade potential. Weathered and calcretized / silcretized horizons (in which the primary features of the kimberlite units are obscured) are present overlying all 3 lobes. Zones of high country rock dilution are also present in all lobes and are referred to as breccias. In addition to these units, the North and Centre Lobes are each infilled by single volumetrically dominant kimberlite units that are texturally similar to each other, while the South Lobe comprises 2 volumetrically dominant units (M/PK(S) and EM/PK(S)) and another 3 volumetrically minor units (Table 1-1).

Table 1-1: Kimberlite units identified in the AK6 kimberlite. Units occurring in more than one lobe (e.g. BBX, WBBX, KBBX, CKIMB, WK) were modelled as separate domains for each lobe (hence the N, C and S suffix) for incorporation into the geological model (Section 7.4).

Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
	CKIMB	CKIMB(N)	Calcretised kimberlite
	FK(N)	FK(N)	Fragmental kimberlite
	KBBX	KBBX(N)	Kimberlite and country rock breccia
	WBBX	WBBX(N)	Weathered country rock breccia
	WK	WK(N)	Weathered kimberlite
Center	BBX	BBX(C)	Country rock breccia
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
	CKIMB	CKIMB(C)	Calcretised kimberlite
	FK(C)	FK(C)	Fragmental kimberlite
	KBBX	KBBX(C)	Kimberlite and country rock breccia
	WBBX	WBBX(C)	Weathered country rock breccia
	WK	WK(C)	Weathered kimberlite
South	BBX	BBX(S)	Country rock breccia
	CBBX	CBBX(S)	Calcretised country rock breccia
	CKIMB	CKIMB(S)	Calcretised kimberlite
	EM/PK(S)	EM/PK(S)	Eastern magmatic/pyroclastic kimberlite
	INTSWBAS	INTSWBAS(S)	Large internal block of basalt
	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite
	WBBX	WBBX(S)	Weathered country rock breccia
	WK	WK(S)	Weathered kimberlite
	WM/PK(S)	WM/PK(S)	Western magmatic/pyroclastic kimberlite
	KIMB1	N/a	Volumetrically minor hypabyssal kimberlite
	KIMB3	N/a	Volumetrically minor hypabyssal kimberlite

The geological model presented in this report (Figure 1-2) is updated from that presented in the previous Technical Report (Oberholzer et al., 2017). Changes include minor revisions to the pipe margin where exposed by mining (all 3 lobes) and significant changes to the pipe shell and internal domain model in the South Lobe based on the results of recent core drilling. The most significant change is the recognition of the EM/PK(S) domain as the volumetrically dominant unit in the South Lobe below ~550 masl.

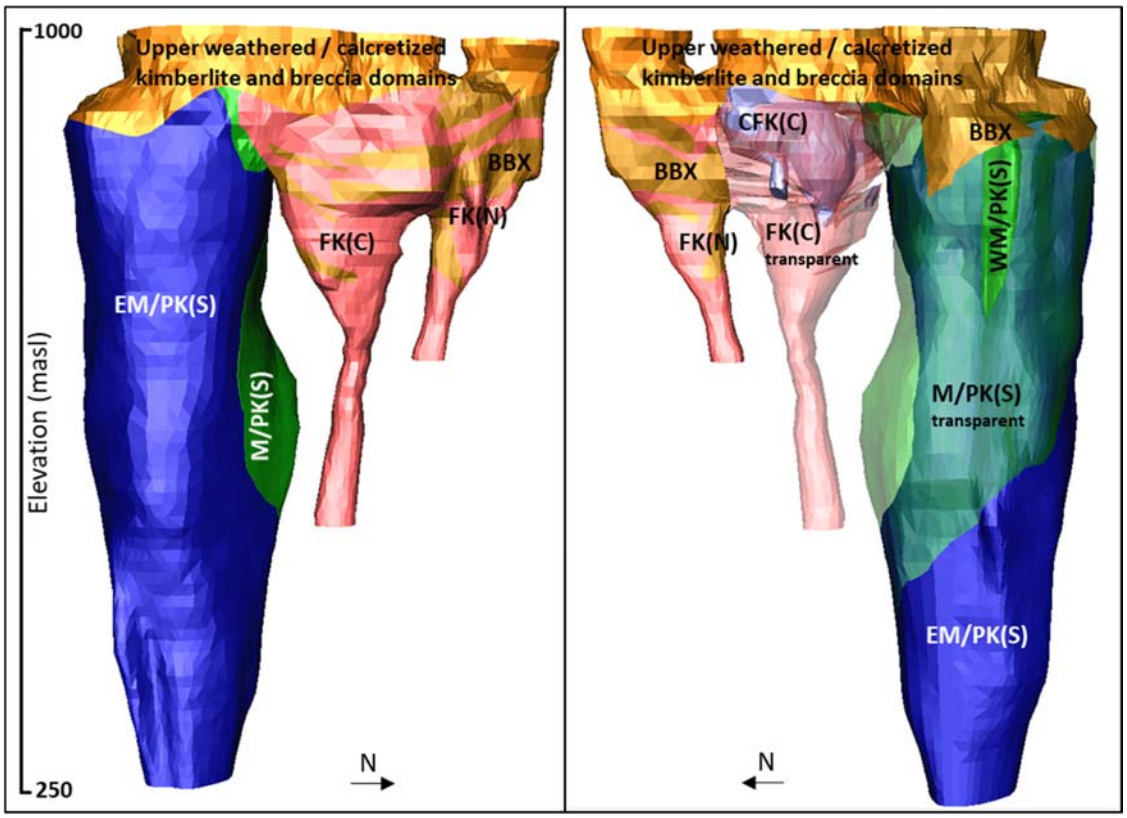

Figure 1-2: Internal geological domains of AK6. The upper 70 to 90 m comprise weathered and calcretized kimberlite and breccia units that are shown with a single colour to simplify the figure; these domains are predominantly mined out, the mine surface as at end December 2017 varies from approximately 60 to 130 mbs. The FK(C) domain in the figure on the right is shown transparent to display the internal CFK(C) domains (purple). The M/PK(S) domain in the figure on the right is shown transparent to display the internal WM/PK(S) domain.

1.4 Exploration, drilling and sampling

AK6 was discovered in 1969 by De Beers. Relevant exploration and evaluation work conducted on the AK6 to date has included:

- Early evaluation and bulk sampling during the period 2003 to 2005;
- Phase 1 advanced exploration (2005 to 2006), including pilot (adjacent to large diameter drill (LDD) holes) and delineation core drilling, LDD drilling / sampling and processing;
- Phase 2 advanced exploration (2006 to 2007), including additional core drilling, LDD sampling and processing and the collection and processing of a large surface trench sample; and
- core drilling and microdiamond[1] sampling in 2017.

[1] The term microdiamond is used throughout this report to refer to diamonds recovered through caustic fusion of kimberlite at a bottom screen size cut-off of 0.105 mm (~0.00002 ct). Rare larger diamonds that may be recovered by a commercial production plant may be recovered through this process but are still referred to as microdiamonds.

Key datasets used as a basis for the Mineral Resource Estimate presented include:

- Core drilling of 61 delineation holes (27,855 m) and 23 pilot holes (4,181 m).
- LDD (23 inch diameter) drilling of 25 holes comprising 7,964 m. The sample dataset generated from these holes comprises 573 samples with a measured volume of 1,924 m^3 (calculated 3,901 tonnes) from which 1,250 ct (larger than DTC1 sieve size) were recovered.
- Processing and analysis of 7,315 kg of drill core (916 individual sample aliquots) for microdiamonds.
- Analysis of 2,808 bulk density samples.
- Mine production records and sales information for all ore processed and diamonds recovered since inception of mining in April 2012. This includes processing results for 13.89 million tonnes of kimberlite, from which 2.21 million carats have been recovered. Sale of diamond production has generated a total of 1.25 billion US$ in revenue.

1.5 Mineral Resource Estimate

The Mineral Resource Estimate for AK6 above 604 masl is restated with minor modifications from the previous project Technical Report (Oberholzer et al., 2017). A high confidence geological model and comprehensive bulk density dataset constrain estimates of volume and tonnage. Grade estimates are based on a well-distributed LDD sample dataset that supports the interpolation of local grade estimates. Modifications to the estimate presented in Oberholzer et al. (2017) include revisions to the geological model, slightly more aggressive capping of outlier grade values used for interpolation and update of diamond values in the South Lobe to reflect the current production and sales dataset.

The Mineral Resource Estimate for AK6 below 604 masl has been significantly revised based on the results of core drilling and microdiamond sampling work carried out in 2017. Volume and tonnage estimates are similarly based on the AK6 geological model and a spatially representative broad bulk density sample coverage. Grade has been estimated using a microdiamond-based approach that is based on a calibration of the ratio of microdiamond stone frequency (stones per kilogram) to + 1 mm LDD macrodiamond[1] grade. The calibration was based on LDD-recovered macrodiamond data and microdiamonds from adjacent pilot hole drill core samples. Drill core microdiamond results (providing a broad spatially representative coverage of the South Lobe below 604 masl) were used, in conjunction with the established ratio of stone frequency to +1 mm LDD grade, to derive average grade estimates for the M/PK(S) and EM/PK(S) domains present below 604 masl in the South Lobe.

The +1 mm (LDD-based) grade estimates above and below 604 masl were adjusted for recovery at a bottom cut-off of 1.25 mm by the Karowe plant in its current configuration.

[1] The term macrodiamond is used throughout this report to refer to diamonds recovered by diamond production plants, which typically only recover diamonds in and larger than the Diamond Trading Company (DTC) sieve category 1 (i.e. > ~0.01 ct).

Diamond values for each lobe are constrained by diamond size frequency distributions (SFDs) defined by selected representative parcels from 6 years of production and active mining. Valuation and sales data from production have been used to define value distributions ($/ct per sieve size class) that have been applied to the SFD models for each lobe to generate average recoverable (+1.25 mm) value estimates ($/ct).

A Mineral Resource statement for AK6 is presented in Table 1-2. The resources reported reflect mining depletion and include all remaining ore and stockpile material as at the end of December 2017. The Mineral Resources are classified at an Indicated level of confidence from surface (current mine level ~940 to 870 masl) to an elevation of 400 masl (depth of 600 m below surface). Deeper additional Mineral Resources (400 to 256 masl) are classified at an Inferred level of confidence. All grades are reported as those recoverable above a 1.25 mm bottom cut-off by the Karowe production plant in its current configuration. Average values also represent "recoverable" values that correlate with the +1.25 mm grades reported. These recoverable grade and value estimates should be adjusted as required to reflect any potential plant modifications or changes in ore metallurgy (e.g. increasing hardness with depth) going forward.

Table 1-2: Mineral Resource statement for the Karowe Mine. The reported resources are those remaining (including stockpile material) as of the end of December 2017. LOM = life of mine, SP = stockpile, Mm^3 = million cubic metres, tpm^3 = tonnes per cubic metre, Mt = million tonnes, cpt = recoverable (+1.25 mm) carats per tonne, Mct = million carats, $/ct = recoverable (+1.25 mm) United States dollars per carat).

Classification	Resource	Volume (Mm^3)	Density (tpm^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	$/ct
Indicated	North Lobe	0.62	2.48	1.54	0.20	13.0	222
	Centre Lobe	1.68	2.57	4.32	0.63	14.6	367
	South Lobe	16.29	2.92	47.63	6.78	14.2	716
	Total	18.59	2.88	53.48	7.62	14.2	674
	LOM SP	1.28	1.85	2.36	0.09	3.8	609
	Working SP	1.05	1.91	2.01	0.20	9.7	661
	Total Stockpile	2.33	1.88	4.37	0.29	6.5	645
	Total Indicated	**20.92**	**2.77**	**57.85**	**7.90**	**13.7**	**673**
Inferred	**South Lobe**	**1.93**	**3.02**	**5.84**	**1.17**	**20.0**	**716**

1.6 Mineral Reserve Estimate

Mineral Reserve Estimate for the open pit portion of the Karowe Mine has been updated based on the updated Indicated Mineral Resource Estimate. Inferred Resources have not been used to estimate Mineral Reserves. The Resource to Reserve conversion was performed by Lucara by conducting an open pit optimisation using Whittle® suite software. The outputs of this process include a mining schedule on which to base plant capacity, waste rock quantities, peak capacities and mining fleet parameters. It should be noted that the Whittle® optimisation is ongoing and is being considered within the feasibility study of the Karowe Underground Project.

The Mineral Reserve Estimate has been classified and reported in accordance with the Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral projects' of June 2011 (the Instrument), updated in 2015 and the classifications adopted by the CIM Council in November 2011.

The effective date of the Mineral Reserve Estimate is May 2018.

The Mineral Reserves (Table 1-3) were derived from the Mineral Resource block model. The Mineral Reserves are the Indicated Mineral Resources that have been identified as being economically extractable through the current open pit mining approach, incorporating mining losses and the addition of waste dilution. The Mineral Reserves form the basis for the open pit mine plan and incorporate stockpiled kimberlite.

Table 1-3: Mineral Reserve Statement for the Karowe Mine.

Open Pit Mineral Reserve Estimate for the Karowe Diamond Mine, Botswana, as at May 2018						
Lobe	Reserve Category	Tonnes (Mt)	Recoverable Grade[13] (cpht)	Recoverable Carats (Mcts)	Diamond Revenue[6] (US$/ct)	Unit Revenue (US$/t)
North	Probable	1.04	13.37	0.14	222	29.68
Centre	Probable	3.37	14.57	0.49	367	53.46
South	Probable	15.43	12.74	1.97	716	91.22
In-situ Reserve (OP Material)[14]		**19.84**	**13.08**	**2.60**	**624**	**81.58**
Working Stockpiles[11]	Probable	2.10	9.96	0.21	661	65.83
LOM Stockpiles[12]	Probable	3.46	4.57	0.16	609	27.84
Total Reserve[15,16]		**25.40**	**11.66**	**2.96**	**625**	**72.95**

Notes:

1. The Mineral Reserve has been depleted for mining up to May 2018
2. Figures have been rounded to the appropriate level of precision for reporting
3. Due to rounding, some columns or rows may not compute exactly as shown
4. The Mineral Reserves are stated as in-situ dry metric tonnes
5. The Mineral Reserves were prepared under the guidelines of the CIM, for reporting under NI 43-101
6. Diamond price is based on diamonds recoverable with current Karowe plant process and Lucara Diamond Price Book
7. Modifying factors for mining recovery of 97 % and waste dilution of 3 % at 0.0 cpht have been applied
8. Probable Mineral Reserves were derived from Indicated Mineral Resources
9. Mineral Reserves are inclusive of Mineral Resources
10. There are no known legal, political, environmental, or other risks that could materially affect the potential Mineral Reserves
11. Working stockpiles comprise surface loose stocks of material with estimated grades exceeding 7 cpht; includes High Grade (HG), Medium Grade (MG), Low Grade (LG) and Contact kimberlite
12. Includes existing LOM Stockpiles of Very Low Grade (VLG) kimberlite material (< 7cpht) as well as in-situ VLG material (currently part of in-situ resource) expected to be directed to the LOM stockpile (1.0Mt @ 6.24 cpht in-situ and 2.5Mt @ 3.9 cpht current surface stocks @ average value of US$ 609/ct). LOM Stockpiles will be processed at the end of life of open pit mining
13. Based on the updated Mineral Resource estimate as presented in this report (1.25 mm bottom cut off size - BCOS) – 70 % of in-situ carats at 1.00 mm BCOS
14. Exclusive of current stockpiles and VLG in-situ material (see note 12 above)
15. Inclusive of current stockpiles and VLG in-situ material (see note 12 above)
16. The Mineral Reserves reported in this table are attributable solely to the ore to be mined (and processed or stockpiled for later processing) from the open pit mine at Karowe

1.7 Conclusion and recommendations

This Technical Report provides an update to the AK6 Mineral Resource Estimate and provides an updated Mineral Reserve statement for the open pit portion of the Karowe Mine. Evaluation work carried out in 2017 has revised and increased confidence in the Mineral Resources present at depth allowing for the classification of previously Inferred Mineral Resources in the elevation range 600 to 400 masl at an Indicated level of confidence. Uncertainty in Mineral Resource Estimates below 400 masl is mostly related to a paucity of drill coverage and corresponding poorer constraints on the pipe shell and internal geology and less representative spatial coverage for microdiamond sampling. Additional core drill coverage and microdiamond sampling would provide a basis for upgraded confidence in this deeper material.

The open pit mining schedule produced from the Whittle® optimisation and the Mineral Reserve estimate have been used as the basis of for a financial model for the project. The financial model indicates that the mine has positive economics to the end of open pit mining, and that the current NPV is USD 480.8 million (at 8 % discount rate).

2. Introduction

The Karowe Mine is an existing open pit diamond mine extracting and processing ore from the AK6 kimberlite. The mine is located in the Central District of Botswana and is part of the Orapa kimberlite field which includes the Orapa, Damtshaa and Letlhakane diamond mines. The Karowe Mine has been in production since April 2012, operated by Boteti Mining (Pty) Ltd. (Boteti), a wholly owned subsidiary of Lucara Diamond Corp (Lucara).

This report has been prepared by Mineral Services Canada Inc. (MSC) in accordance with the reporting requirements stipulated by National Instrument 43-101 (NI 43-101) standards for disclosure of mineral projects in Canada. Unless otherwise stated, all monetary figures expressed in this report are in United States dollars (US$) and all units are in metric measures. The coordinate systems used are Universal Transverse Mercator (UTM) in the datum WGS84 and zone 35S or geographic latitude and longitude expressed as decimal degrees with true North bearings in the datum WGS84. A list of all abbreviations used is provided prior to the executive summary of this report.

The report has been compiled by Mineral Services Canada Inc. with contributions by Lucara Diamond Corp. and Lofty Mining (Pty) Ltd (Sections 1.6 and 16 to 22). Much of the report is restated and summarised from Oberholzer et al. (2017).

2.1 Scope of work

This report provides an update to the previous Mineral Resource Estimate for AK6 (Oberholzer et al., 2017), which stated Indicated Mineral Resources from surface (~1000 masl) to an elevation of 600 masl and Inferred Mineral Resources from 600 masl to the base of the geological model at 256 masl. Exploration work carried out in 2017 (core drilling and sampling) was focussed on the deep portion of the body below

600 masl and has resulted in a revision of the geological model at depth and increased confidence in the updated volume and grade estimates at depth, allowing for an extension of the Indicated Mineral Resource to an elevation of 400 masl. This report provides details of all recent (previously unreported) exploration work and documents the update to the Mineral Resource Estimate. Estimates above 600 masl are restated from Oberholzer et al. (2017) with minor modifications as detailed in Section 14.

2.2 Previous Technical Reports

The following Technical Reports for the AK6 kimberlite / Karowe Mine are available on www.sedar.com:

- Oberholzer, G.J., Blackham, N.G.C, Cox, J.A., Thompson, J.J., Morton, K.L, Nowicki, T., Armstrong, J., 2017: NI 43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project, Effective Date 31 October 2017.
- Lynn M.D., Nowicki 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 Qualified Persons (as defined in NI 43-101) responsible for each of the sections of this Technical Report are listed in Table 2-1. Certificates for Qualified Persons are attached in Section 28.

Dr Tom Nowicki has 25 years of 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. (MSC) 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”. Dr Nowicki visited the mine on the 3rd and 4th July 2013.

Mr. Lofty Julius Hendrik (Henk) Fourie of Lofty Mining (Pty) Ltd is a mining engineer with over 35 years of experience. Mr Fourie is a Professional Engineer (Pr Eng) in good standing with the Engineering Council of South Africa (ECSA), and has the relevant qualifications, experience, competence and independence to act as a “Qualified Person”.

Dr John Armstrong has over 28 years of combined experience in mineral exploration, mining and government and is a registered professional geoscientist in good standing with the Association of Professional Engineers and Geoscientists of the Northwest Territories and Nunavut (NAPEG). 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. (Lucara) since

September 2013 and implemented predictive size distribution models for the Karowe Mine supporting the presence and recovery of large diamonds. Dr Armstrong is not independent of Lucara.

Table 2-1: Qualified Persons responsible for each of the sections of this Technical Report.

Qualified Person	Qualification	Professional registration	Sections
Dr Tom Nowicki	PhD	P.Geo. (APEGBC)	1.1 to 1.5; 1.7; 2 to 14; 23 to 28
Dr John Armstrong	PhD	P.Geo. (NAPEG)	16.1; 16.2; 17 to 22
Lofty Julius Hendrik Fourie	B.Eng	Pr Eng (ECSA)	1.6; 15; 16.3; 16.4

2.4 Principal sources of information

MSC has based its review on information provided by Lucara along with technical reports by previously engaged consulting firms and other relevant published and unpublished data.

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

This report has been prepared on information available up to and including April 2018. MSC has provided consent for the inclusion of this Independent Technical Report in public disclosure documents.

3. Reliance on other experts

MSC has not independently verified (and is not qualified to verify) the legal status of the Mining Licence (Section 4.1.3) that forms the subject of this report, or of associated permits (Section 4.2.5). MSC has relied on confirmation from Lucara (via Dr Armstrong) that all permits are valid and in good standing as per the information provided. Various people associated with and contracted by Lucara Diamond Corporation and Karowe Diamond Mine have made contributions to technical detail outlined in this report as indicated in the sections below.

4. Property description and location

The contents of Section 4 are extracted verbatim from Oberholzer et al. (2017). Table 4-2 has been updated to reflect current permit status.

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 license 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 4.2.2).
- 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.

4.1.3 Issuer's Title, Location and Demarcation of Mining License

The property is Mining Licence (ML) 2008/6L issued in terms of the Mines and Minerals Act 1999, Part VI, and covering 1,523 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. An aerial photograph of the mine is shown in Figure 4-3.

Table 4-1: Corner point locations of Mining License 2008/6L. Datum WGS84.

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

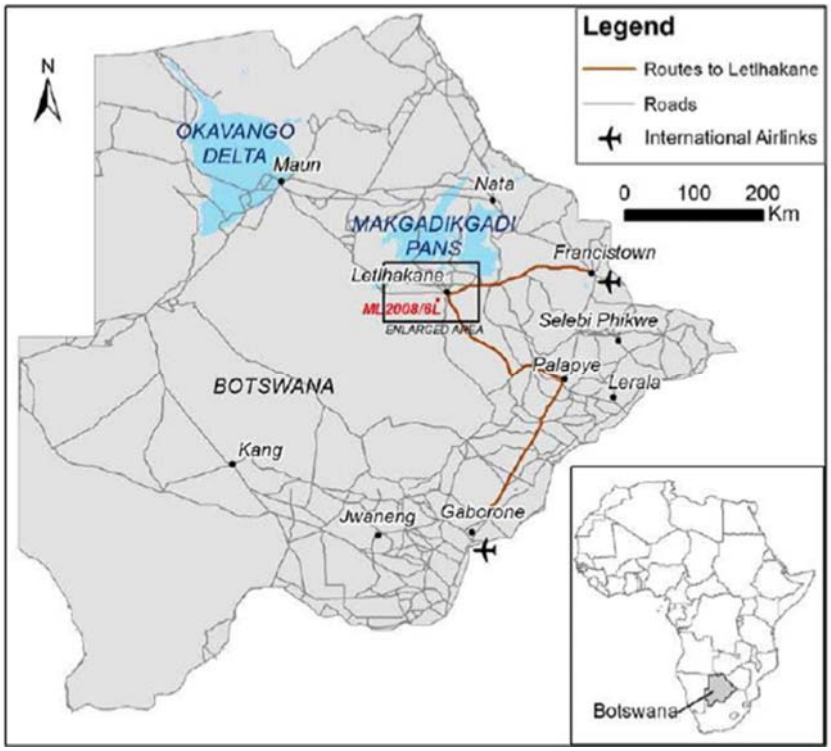

Figure 4-1: Regional locality map.

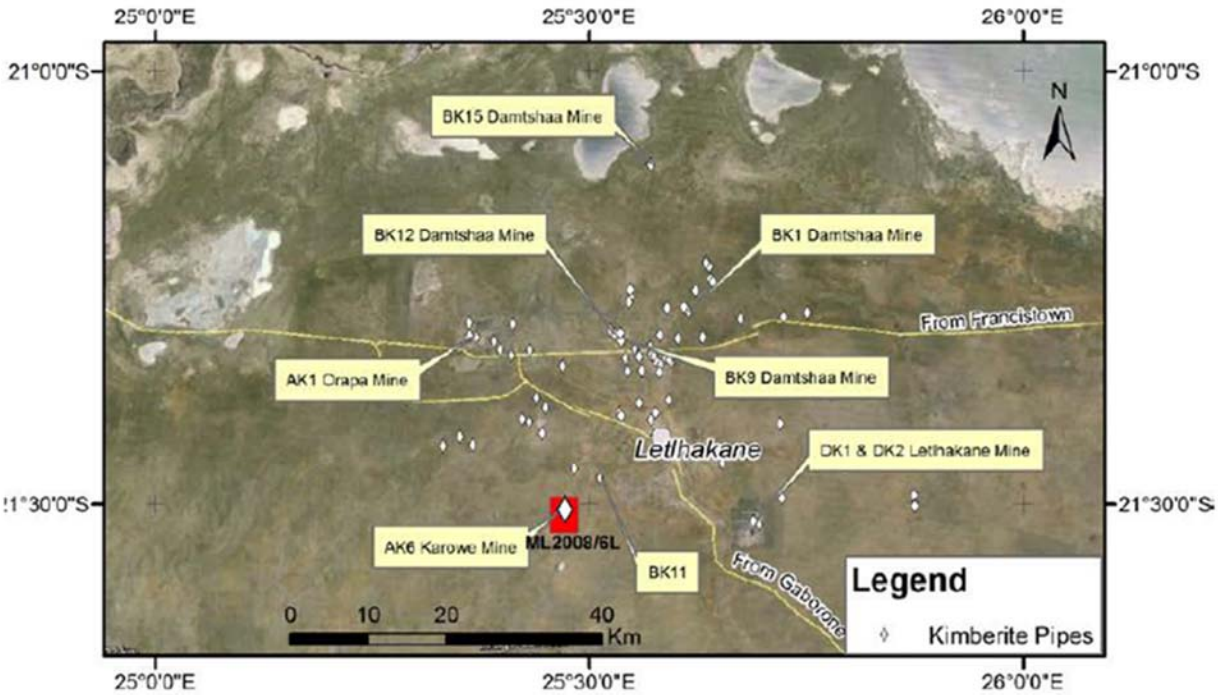

Figure 4-2: Karowe Mine location and other diamond mines in the vicinity.

Figure 4-3: Aerial photograph of the Karowe Mine. The photograph is 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.

4.2 Permitting Rights and Agreements Relating to Karowe Mine

4.2.1 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.

4.2.2 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.

4.2.3 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.

4.2.4 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.

4.2.5 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/649/07-2080/17 - 003 Kellinicks CRLIC/450/08-1881/18 - 008 Modi mode CRLIC/450/09-1881/17 - 003 Modi mode CRLIC/450/06-1881/17 - 004 Modi Mode CRLIC/450/06-1881/17-009 Modi- Mode CRLIC/01/04-063/17- SKIP HIRE CRLIC/01/03-063/17 – SKIP HIRE	31/08/2018 31/07/2018 30/09/2018 30/06/2018 30/06/2018 30/04/2018 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	BW0315/2017	Renewed and certificates will expire in November 2019	Radiation Inspectorate	Radiation Protection Act
Waste Facilities & Sewage Plant	Application in Progress	The mine is working on two projects both at the landfill and Sewage plant to address the findings of the Department of Waste Managemnet and Pollution Control	Dept. of Waste Management and Pollution Control	Waste Management Act
License to manufacture explosives	In Place	31/12/2018	Dept. of Mines	Explosives Act
Permit to carry bulk explosives	F35/13, F34/13 and F36/13	31/12/2018	Dept. of Mines	Explosives Act
Magazine License	386:00002948A and 385:00002947A	31/12/2018	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

The contents of Section 5 are extracted verbatim from Oberholzer et al. (2017).

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 masl 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). At present the population therefore may be on the order of 35,000 to 40,000. 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 1800 m 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: Average monthly temperature and rainfall at the Karowe Mine.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average temperature (^{o}C)	24.6	24	23	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 contents of Section 6 are extracted from Oberholzer et al. (2017) and have been updated as necessary to reflect currently available information.

The AK6 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 led to a series of other kimberlite discoveries in the Orapa region. Commercial production at Karowe 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 are 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 cpm^3 (approximately 3.5 cpht; this sampling was not NI 43-101 compliant).

One vertical cored borehole was drilled into the kimberlite to a depth of 61 m with weathered primary kimberlite recorded from a depth of 8 m (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 AK6 kimberlite and Karowe Mine lies within former prospecting license 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 AK6 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 cpm^3 (approximately 15 cpht; not NI 43-101 compliant).

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

PL 1/97 was issued to De Beers Prospecting Botswana (Pty) Ltd. (Debot) on 1 February 1997 and covered the AK6 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 PL 13/2000 with an area of 9.95 km^2 over the AK6 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 AK6, 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. This included PL 13/2000 and AK6. 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 PL 13/2000 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 28 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 9 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 28 October 2008.

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

6.7 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 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 Q1 2018 (Table 6-1) the Karowe Mine has produced 2.21 million ct from 13.89 million tonnes of processed kimberlite and has sold via tender a total of 2.06 million carats for a total of US$1.25 billion resulting in an achieved sold average price US$606/ct.

Table 6-1: Karowe Mine production and sales results.

Year	2012	2013	2014	2015	2016	2017	Q1 2018	Total
Kimberlite mined (tonnes)	1,600,971	3,944,343	3,327,754	2,358,657	2,722,375	1,575,052	630,242	16,159,394
Waste mined (tonnes)	4,074,196	5,493,445	10,270,720	11,407,010	11,058,041	15,865,121	3,991,648	62,160,181
Kimberlite processed (tonnes)	1,327,682	2,354,538	2,421,506	2,238,975	2,613,217	2,335,550	599,407	13,890,875
Carats recovered	294,167	440,751	430,292	365,690	353,974	249,767	75,698	2,210,339
Recovered grade (cpht)	22	19	18	16	14	11	13	16
Carats sold	152,724	438,717	412,136	377,136	358,806	260,526	63,317	2,063,362
Sales average $/ct	$274	$415	$617	$612	$824	$847	$401	$606

In mid-November 2015 the Karowe Mine recovered the world's second largest diamond gemstone, the 1,109 ct Lesedi La Rona. The following day the Karowe Mine recovered the 813 ct 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 Q1 2018

From inception to the end of Q1 2018 a total of 158 diamonds have sold for greater than US$1.0M a piece and over 77 gem quality diamonds greater than 100 ct have been sold as individual stones. In the same time period the Karowe Mine has recovered 7 diamonds >300 ct, 26 diamonds between 200 and 300 ct and an additional 112 diamonds between 100 and 200 ct.

7. Geological setting and deposit geology

A detailed account of the geological setting and geology of the Karowe Mine was provided in Lynn et al. (2010). A summarised version was provided in the previous Technical Report (Oberholzer et al., 2017). The summarized version has been restated here, with additional details and updates provided in Sections 7.3 and 7.4 to document a major revision to the geological model for the deep portion of the South Lobe based on the results of core drilling carried out in 2017.

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 (OKF), the bedrock may be overlain by up to 40 m of Kalahari Group sediments.

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 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 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 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 of local geology.

7.2 Property geology

Drilling has defined the country rock succession at the Karowe Mine property as shown in Table 7-1. The volcanic and sedimentary units are almost flat lying.

Table 7-1: Stratigraphic thicknesses at the Karowe Mine property.

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

7.3 Kimberlite geology

The geological summaries presented in this section are extracted and summarized from internal De Beers documentation (Stiefenhofer, 2007; Hanekom et al., 2006; Tait and Maccelari, 2008) and from a more recent report documenting review, core logging and petrography work carried out by MSC (MSC18/005R).

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 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 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 material has been grouped into mappable units (Table 7-2) based on geological characteristics and interpreted grade potential, including separation of internal portions of the pipe with very high country rock xenolith dilution (referred to historically as breccias). 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, MSC18/005R). The main geological features of each unit are summarised below.

Table 7-2: Kimberlite units identified in the AK6 kimberlite. Units occurring in more than one lobe (e.g. BBX, WBBX, KBBX, CKIMB) were modelled as separate domains for each lobe for incorporation into the geological model (Section 7.4). Volumetrically minor units (KIMB1, KIMB3) that could not be resolved as discrete domains were incorporated into other South Lobe domains as explained in Section 7.4.2.

Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
	CKIMB	CKIMB(N)	Calcretised kimberlite
	FK(N)	FK(N)	Fragmental kimberlite
	KBBX	KBBX(N)	Kimberlite and country rock breccia
	WBBX	WBBX(N)	Weathered country rock breccia
	WK	WK(N)	Weathered kimberlite
Center	BBX	BBX(C)	Country rock breccia
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
	CKIMB	CKIMB(C)	Calcretised kimberlite
	FK(C)	FK(C)	Fragmental kimberlite
	KBBX	KBBX(C)	Kimberlite and country rock breccia
	WBBX	WBBX(C)	Weathered country rock breccia
	WK	WK(C)	Weathered kimberlite
South	BBX	BBX(S)	Country rock breccia
	CBBX	CBBX(S)	Calcretised country rock breccia
	CKIMB	CKIMB(S)	Calcretised kimberlite
	EM/PK(S)	EM/PK(S)	Eastern magmatic/pyroclastic kimberlite
	INTSWBAS	INTSWBAS(S)	Large internal block of basalt
	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite
	WBBX	WBBX(S)	Weathered country rock breccia
	WK	WK(S)	Weathered kimberlite
	WM/PK(S)	WM/PK(S)	Western magmatic/pyroclastic kimberlite
	KIMB1	N/a	Volumetrically minor hypabyssal kimberlite
	KIMB3	N/a	Volumetrically minor hypabyssal kimberlite

7.3.1 Units defined by weathering and country rock dilution

Certain kimberlite units have been classified based on alteration and weathering characteristics which obscure the primary features of the kimberlite. Zones of very high country rock dilution (either in situ brecciated country rock with minor intruded kimberlite or zones of high xenolith content within the pipe) have historically been referred to as breccias. This terminology has been maintained for continuity with previous reporting. These breccia, weathered and calcretised units occur in all three lobes of the pipe and are described below. Note that the geological domain models representing these units have been separated by lobe (Table 7-2).

Calcretised kimberlite (CKIMB)

The upper parts of all three lobes contain severely calcretised and silcretised rock. This zone is typically ~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 the top of all three lobes (Opperman and van der Schyff, 2007).

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 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. Separate weathered units have therefore been defined in each lobe for each of the geology domains where weathered equivalents of these domains are present at surface. Separate models of these units are required as weathering has significant implications for the metallurgical properties of kimberlite.

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 %). These basalt breccias consist of large (meter-sized) to smaller basalt clasts set in a matrix of kimberlite. The bulk of the breccias occur close to the wall-rock contacts in each lobe. 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 units. 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).

7.3.2 North Lobe kimberlite units

FK(N) – Fragmental kimberlite

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.

7.3.3 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 non-fragmental 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, as described below.

CFK(C) – Carbonate-rich fragmental kimberlite

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 void space, 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

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.

7.3.4 South Lobe kimberlite units

The upper, western part of the South Lobe is dominated by weathered kimberlite (WK(S)), a weathered basalt breccia (WBBX(S)), an underlying unaltered basalt breccia unit (BBX(S)) and a large block (floating reef) of solid basalt (INTSWBAS) recognised and mapped during mining activities in 2013 (Lynn et al., 2014). In addition to these weathered and breccia units an additional 5 units have been recognised. Four of these (M/PK(S), EM/PK(S), KIMB1 and KIMB3) were described in detail during an update of the South Lobe internal geology model in 2018 and the descriptions provided below have been extracted and summarised from MSC18/005R. Descriptions of the unit WM/PK(S) are unchanged from Oberholzer et al. (2017).

M/PK(S) – Magmatic/pyroclastic kimberlite

M/PK(S) is an olivine-rich, country rock xenolith-poor, groundmass-supported, poorly-sorted and broadly massive macrocrystic apparent coherent kimberlite. Macroscopically the kimberlite is grey or grey-green in colour and exhibits a 'black spotted' appearance imparted by the presence of common kelyphitised garnet macrocrysts and black altered phlogopite macrocrysts. Although broadly massive, crude stratification is variably apparent in the form of diffuse fluctuation in olivine and country rock xenolith size and abundance, as well as preferentially oriented elongate components (olivine, small basalt xenoliths and phlogopite macrocrysts). Olivine ranges in size from ultra fine (<0.125 mm) to ultra coarse (> 16 mm) and is predominantly fresh, very abundant (45-50 %) and closely packed. The coarser crystals are inhomogeneously distributed and commonly broken, features atypical of most hypabyssal kimberlite. The groundmass comprises fresh or serpentinised monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. A distinct population of thermally metasomatised/altered country rock xenoliths comprises mainly basalt (grey-green larger clasts and small white elongate shards), lesser white basement granite-gneiss clasts and minor Karoo sedimentary rocks. The overall proportion of crustal dilution is low (typically <10 %), rarely ranging up to a maximum of 25 %. Ilmenite is notably abundant and characterised by variably developed grey reaction rims (comprising fibrous kelyphite-like material). In addition to garnet, ilmenite and rare chrome diopside, the kimberlite contains orthopyroxene xenocrysts with variably developed reaction rims. M/PK(S) is characterised by a relatively high magnetic susceptibility (19 to 30 x 10^{-7} SI).

The high abundance and inhomogeneous distribution of olivine and high proportion of angular olivine grains, combined with the presence of crude stratification and rare probable relict melt-bearing pyroclasts, suggest that M/PK(S) was formed extrusively, and can be described as having a clastogenic or apparent coherent texture.

The name M/PK(S) applied to this unit reflects the initial uncertainty with respect to textural classification of the kimberlite. The kimberlite exhibits textures consistent with a magmatic kimberlite (MK), now referred to as coherent kimberlite (Scott Smith et al., 2013), but also exhibits subtle textures suggesting a possible pyroclastic origin (PK). The M/PK(S) unit is the volumetrically dominant South Lobe infill above ~550 masl. Typical M/PK(S) is shown in core, polished slab and photomicrograph in Figure 7-2.

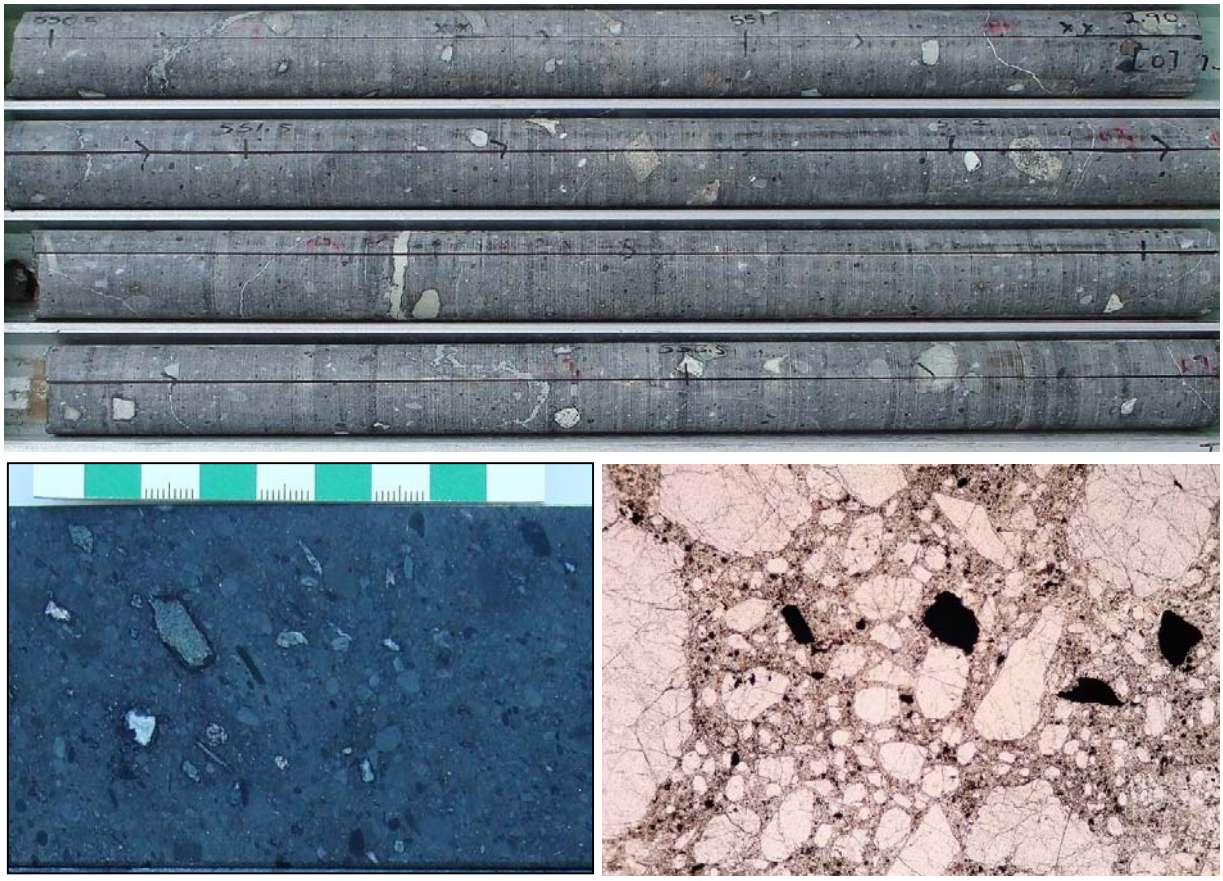

Figure 7-2: Typical appearance of M/PK(S) in HQ drill core (above, hole REP001 from 550 to 554 m), in polished slab (below left, hole REP002 at 639.81 m) and in photomicrograph (below right, hole REP001 at 628.3 m, 2X magnification, PPL, FOV = 7 mm). M/PK(S) – Eastern diluted magmatic/pyroclastic kimberlite

EM/PK(S) is an olivine-rich, country rock xenolith-poor to -rich, groundmass-supported, poorly-sorted and broadly massive macrocrystic apparent coherent kimberlite. Macroscopically the kimberlite is grey-green in colour with variably abundant white 'speckles'. It exhibits a more 'granular' appearance than M/PK(S) due to the olivine being more readily discerned. It lacks the 'black spotted' appearance of M/PK(S) as kelyphitised garnet is less common and phlogopite macrocrysts are fresh. Although broadly massive, crude stratification is variably apparent in the form of diffuse fluctuation in olivine and country rock xenolith size and abundance; preferential orientation of elongate components is rare. Olivine ranges in size from ultra fine (<0.125 mm) to ultra coarse (>16 mm) and is predominantly fresh, very abundant (45-50 %) and closely packed. The coarser crystals are inhomogeneously distributed and commonly broken, features atypical of most hypabyssal kimberlite. The groundmass comprises serpentinised monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. Groundmass spinel is less abundant than in M/PK(S) and generally occurs as single sub/euhedral crystals; crystal aggregates are comparatively rare or absent. The country rock xenolith population differs from M/PK(S) in terms of the relative proportions, appearance and size

distribution of rock types. Basalt is similarly the dominant xenolith type but it occurs as tan-coloured larger clasts and as a distinct population of small (<1 cm) equant tan or grey-green clasts. Karoo sedimentary rock clasts are more abundant than granite-gneiss xenoliths and commonly exhibit zonal alteration and irregular clast margins. The small (<1 cm) white 'speckles' characteristic of this unit are round carbonate fragments (possibly amygdales derived from disaggregated basalt). The thermal metasomatism/alteration assemblage of country rock xenoliths in EM/PK(S) includes common clinopyroxene. The overall proportion of crustal dilution is low (typically <10 %), rarely ranging up to a maximum of 25 %. Ilmenite is similarly characterised by variably developed reaction rims but its abundance is roughly half that of M/PK(S). Orthopyroxene xenocrysts are more common than in M/PK(S) with less well developed reaction rims. EM/PK(S) generally has a lower magnetic susceptibility than M/PK(S) (1.5 to 14 x 10^{-7} SI).

The high abundance and inhomogeneous distribution of olivine and high proportion of angular olivine, combined with the presence of crude stratification and rare probable relict melt-bearing pyroclasts, suggest that EM/PK(S) was formed extrusively, and can be described as having a clastogenic or apparent coherent texture.

The name EM/PK(S) applied to this unit reflects the initial uncertainty with respect to textural classification of the kimberlite as for M/PK(S) described above. The kimberlite exhibits textures consistent with a magmatic kimberlite (MK), now referred to as coherent kimberlite (Scott Smith et al., 2013), but also exhibits subtle textures suggesting a possible pyroclastic origin (PK). The EM/PK(S) unit is the volumetrically dominant South Lobe infill below ~550 masl. Typical EM/PK(S) is shown in core, polished slab and photomicrograph in Figure 7-3.

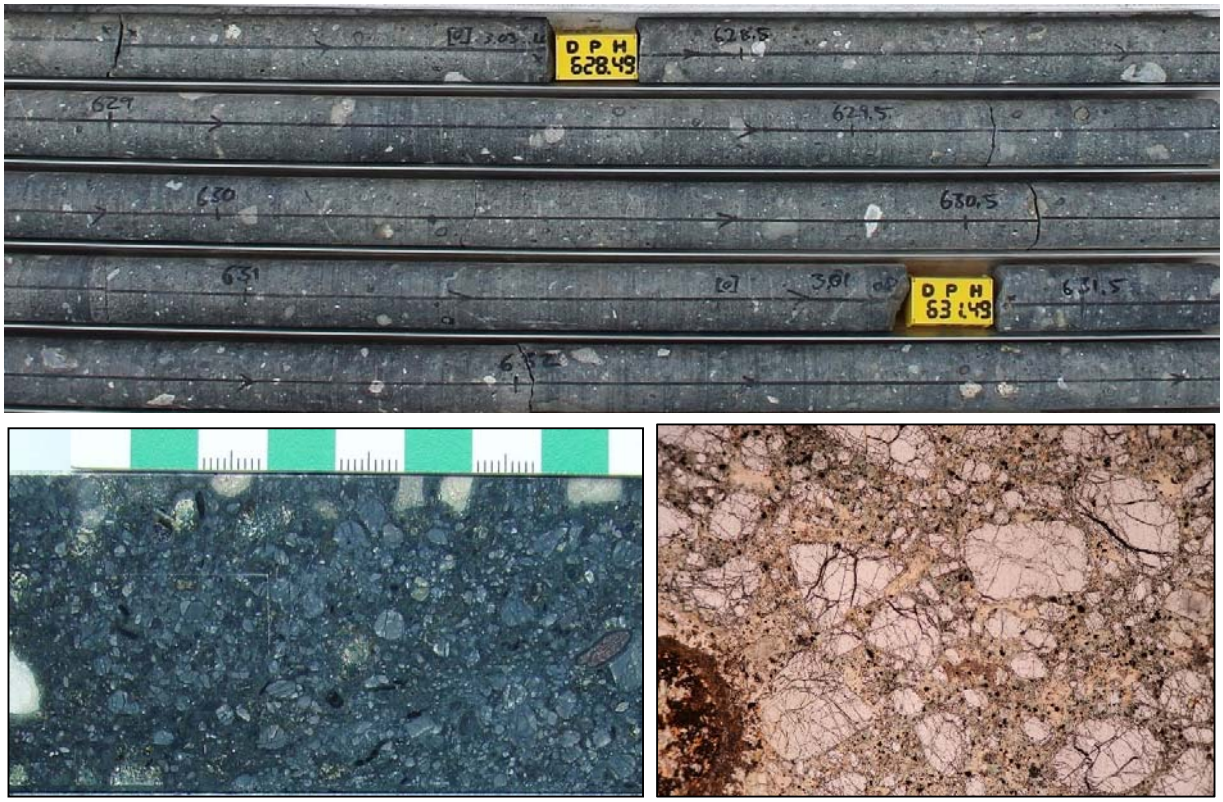

Figure 7-3: Typical appearance of M/PK(S) in NQ drill core (above, hole GT001a from 628 to 632.5 m), in polished slab (below left, hole DDH018 at 301.52 m) and in photomicrograph (below right, hole REP003 at 588.58 m, 2X magnification, PPL, FOV = 7 mm).

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

The WM/PK(S) is a pipe-shaped internal kimberlite unit defined in the western portion of the South Lobe that displays geological characteristics apparently 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.

Minor South Lobe kimberlite units

Two additional units have been identified during recent core logging and petrographic study in the South Lobe (MSC18/005R). These units, referred to as KIMB1 and KIMB3, are a volumetrically minor component (<10 %) of the South Lobe infill, and due to their sporadic and dispersed occurrence cannot be mapped in

as contiguous units as a basis for discrete geological domains. These units were incorporated into the surrounding M/PK(S) and EM/PK(S) units in the construction of the internal domain model.

KIMB1 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor massive to locally flow-aligned macrocrystic hypabyssal kimberlite. Macroscopically the kimberlite is dark grey-black in colour with readily discernible olivine ranging in size to ultra coarse (> 16 mm). Olivine distribution is more uniform than in M/PK(S) and EM/PK(S) and broken crystals are present but notably less common. The groundmass comprises abundant phlogopite as ultra fine-grained tablets, which contrasts with the poikilitic plates in M/PK(S) and EM/PK(S), as well as lesser monticellite, perovskite, spinel, serpentine/ chlorite and carbonate. Crustal dilution is typically low (<5 %) and includes basalt, granite-gneiss and Karoo sedimentary rock clasts in variable relative proportions. Both fresh and completely kelyphitised garnet are common and ilmenite generally lacks reaction rims like those typical of M/PK(S) and EM/PK(S). Fresh garnet lherzolite and other mantle xenoliths are common. Phlogopite macrocrysts are either fresh or partially altered along crystal margins (leaving the cores fresh). Rare autoliths of unknown origin occur locally. Contacts with M/PK(S) and EM/PK(S) are typically abrupt yet diffuse in detail, and in rare instances are sharp with finer-grained flow zones. Together these features suggest KIMB1 represents low-volume late-stage intrusions emplaced after the main pipe filling units, possibly in some cases before the host units were completely consolidated. Magnetic susceptibility readings for KIMB1 are highly variable but most commonly less than 20 x 10^{-7} SI.

KIMB3 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor, massive macrocrystic hypabyssal kimberlite. Macroscopically the kimberlite is dark grey-green in colour and characterised by readily discernible altered olivine (typically with dark margins) ranging in size to ultra coarse (> 16 mm). Olivine distribution is more uniform than in M/PK(S) and EM/PK(S) and broken crystals are rare. Olivine macrocryst abundance is lower than in KIMB1, M/PK(S) and EM/PK(S). The groundmass displays a variably segregationary texture and comprises acicular to prismatic decussate non-pleochroic phlogopite laths, minor serpentinised monticellite, perovskite, spinel (including common atoll textured), serpentine/ chlorite, carbonate and abundant hydrogarnet. Crustal dilution is typically very low (<5 %) and includes mainly basalt and granite-gneiss. Garnet is either partly fresh or completely kelyphitised and ilmenite generally lacks reaction rims like those typical of M/PK(S) and EM/PK(S). Garnet, ilmenite and mantle xenoliths are generally present in lower abundances than the other units. Phlogopite macrocrysts are typically completely altered. Rare autoliths of unknown origin occur locally. Contacts with M/PK(S) and EM/PK(S) are typically abrupt yet diffuse in detail, and in rare instances are sharp with finer-grained flow zones. Together these features suggest KIMB3 represents low-volume late-stage intrusions emplaced after the main pipe filling units, possibly in some cases before the host units were completely consolidated. Magnetic susceptibility readings for KIMB3 are highly variable but in general are the highest of all the units, commonly ranging between 20 and 60 x 10^{-7} SI.

7.4 AK6 geological model

The geological model of AK6 consists of two components: (1) a pipe shell model, representing the morphology and extent of the deposit, and (2) an internal geological domain model, made up of multiple solids constructed to represent the spatial distribution of kimberlite and other domains. The pipe shell model has been updated (MSC17/006R and MSC18/004R) from that reported in Oberholzer et al. (2017) for recent mining exposure of the contact (all lobes) and at depth in the South Lobe to honour pierce point information from core drilling carried out in 2017. The internal domain model for the Centre and North Lobes remains unchanged from that documented in Oberholzer et al. (2017). The South Lobe internal domain model has been significantly revised (below the boundary between weathered and unweathered kimberlite) based on logging of new deep core drill holes, petrography and photo review / drill log update of historical drill cores (MSC18/005R). The two main updates are (1) a substantial increase in the size and change in the shape of the EM/PK(S) domain corresponding with a significant decrease in the size of the M/PK(S) domain, and (2) the discarding of the YIELD17 domain. The YIELD17 domain was a sub-domain of M/PK(S) that was defined on the basis of very high DMS yields from LDD sample processing and was therefore a metallurgical domain and did not represent a distinct geological unit. Due to recent plant upgrades (Section 13) it is no longer necessary to distinguish this characteristic, and the previous YIELD17 domain (Oberholzer et al., 2017) has been included with M/PK(S) in this update.

7.4.1 Shell model

The 2018 pipe shell model is defined by a total of 154 pierce points in 71 core drill holes and an additional 16 pierce points in 13 LDD holes (certain holes provide 2 pierce points, entering and leaving the pipe). Additional information on minimum shell constraints are provided by the substantial internal LDD and core drill coverage. The shell extends from surface (~1000 masl) to a minimum elevation of 256 masl (Figure 7-4). The degree of control on the pipe shell is relatively high down to 350 masl. Below 350 masl the shell model is based on a single pierce point and downward continuation of established wall rock dips.

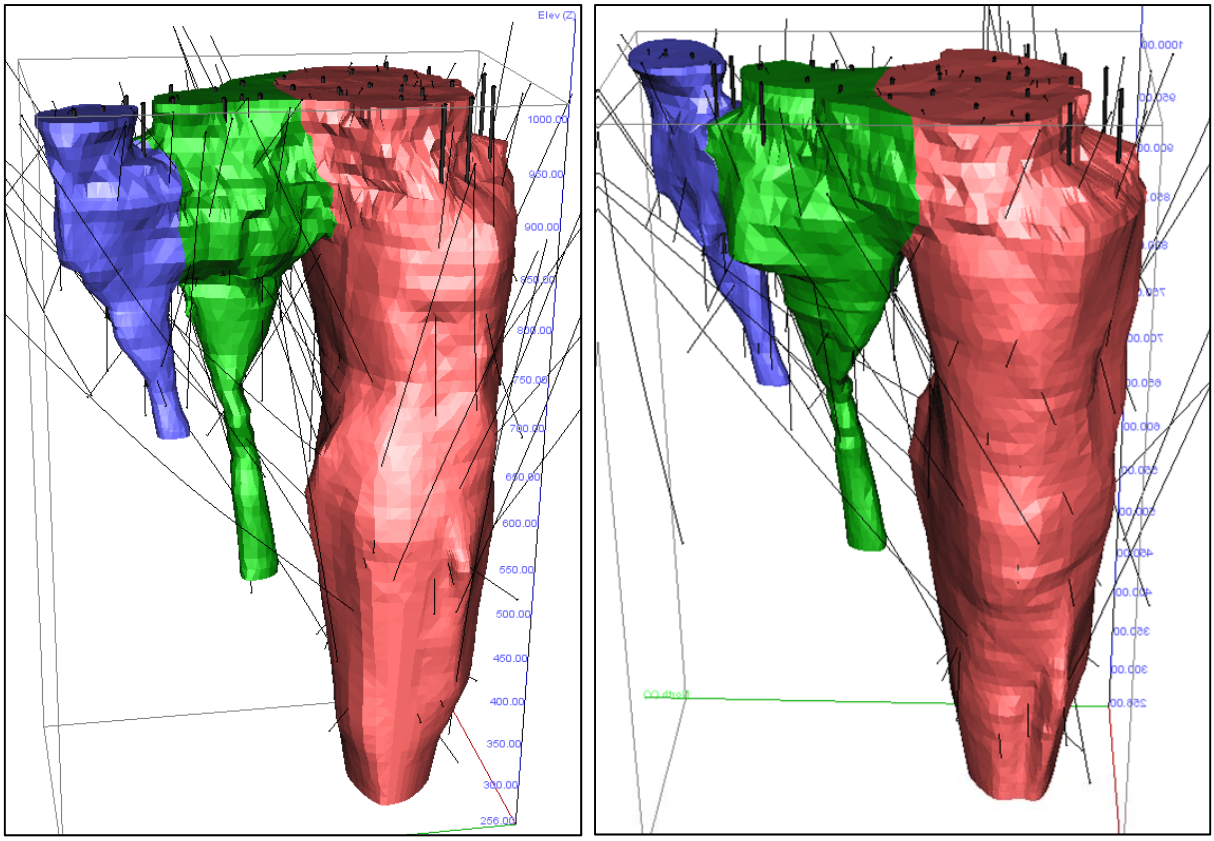

Figure 7-4: AK6 shell model, colour coded by Lobe (blue = North, green = Centre and red = South), and all drill core (thin black trace) and large diameter drill (thick black trace) coverage. View is oriented towards the ESE in the left figure and to the west in the right figure.

7.4.2 Internal domain model

The internal geological domain model comprises a series of wireframe triangulation solids representing the spatial distribution of the various kimberlite units within the 3 lobes (Table 7-2). No changes have been made to the internal domain boundaries reported in Oberholzer et al. (2017) for the North and Centre Lobes, and for the South Lobe above the base of the weathering horizon (~910 to 930 masl). The internal geological domains are shown in Figure 7-5.

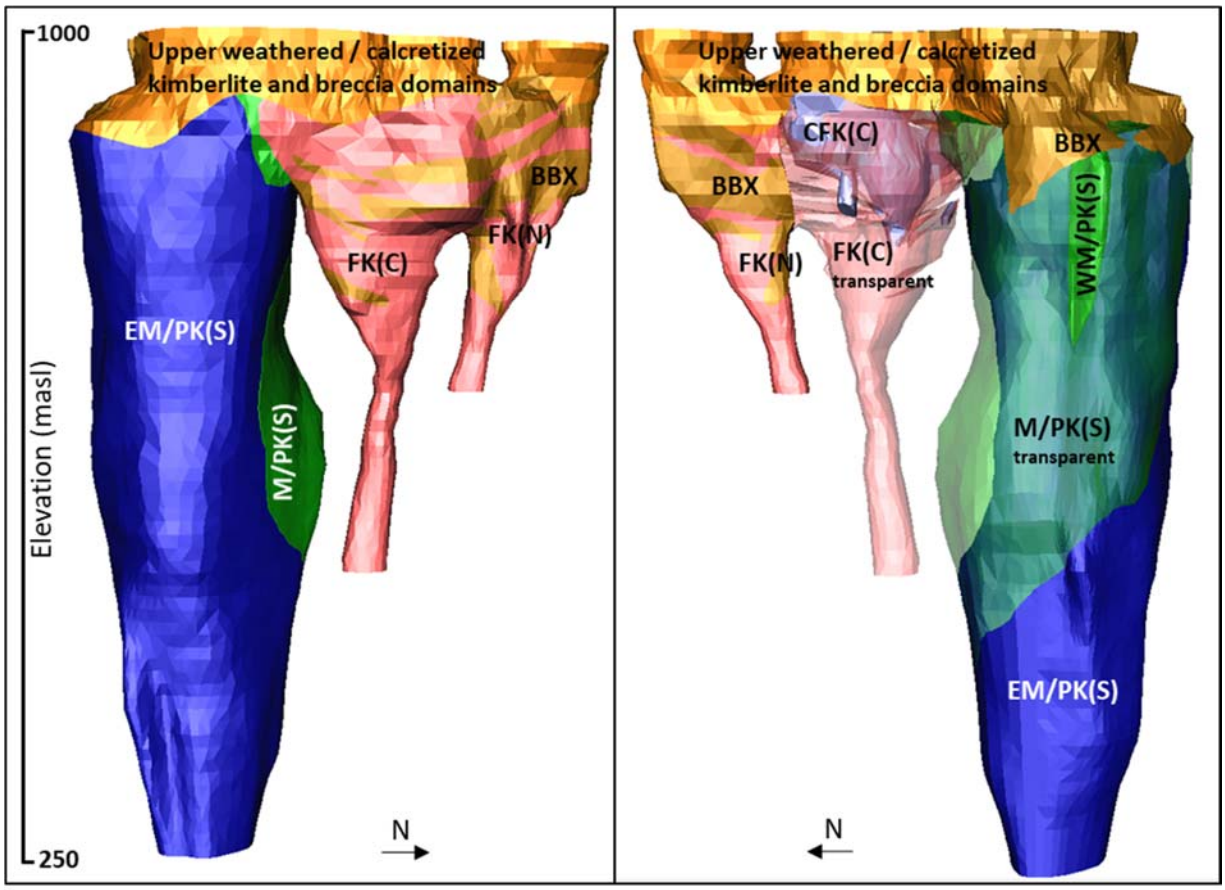

Figure 7-5: Internal geological domains of AK6. The upper 70 to 90 m comprise weathered and calcretized kimberlite and breccia units that are shown with a single colour to simplify the figure; these domains are predominantly mined out, the mine surface as at end December 2017 varies from approximately 60 to 130 mbs. The FK(C) domain in the figure on the right is shown transparent to display the internal CFK(C) domains (purple). The M/PK(S) domain in the figure on the right is shown transparent to display the internal WM/PK(S) domain. The morphologies of the M/PK(S) and EM/PK(S) domains are better illustrated in Figure 7-6.

The M/PK(S) and EM/PK(S) model solids have been significantly revised from those reported in Oberholzer et al. (2017). Comparison of 2017 drill core geology to the 2017 geological model suggested the extent of the EM/PK(S) domain had previously been underestimated and a photographic review of historical South Lobe drill cores was carried out to support the development of a more accurate EM/PK(S) domain model. The revised extent of the EM/PK(S) domain based on this work indicated the probable presence of EM/PK(S) at the current pit surface and the contact between the M/PK(S) and EM/PK(S) domains was subsequently mapped where exposed and accessible. This was undertaken by Boteti geologists using the diagnostic macroscopic characteristics of the kimberlite units and their typical ranges in magnetic susceptibility (as per MSC17/038R). This partially mapped contact was incorporated into the domain model and supported the collection of a controlled production bulk sample (Section 14.4.1) of EM/PK(S). A list of the domain model solids and their volumes is provided in Table 14-1 in Section 14.2.1, as the

volumes of these domains form the basis for the Mineral Resource tonnage estimates provided in Section 14.3. The number and length of core drill holes intersecting each domain are given in Table 7-3. The morphologies of the M/PK(S) and EM/PK(S) domains and the internal drill coverage on which they are based are illustrated in Figure 7-6.

Table 7-3: Core drill coverage of internal geological model domains.

Lobe	Domain model solid	Number of holes intersecting solid	Drill hole intersection length (m)
North	BBX1(N)	9	297.64
	BBX2(N)	1	9.99
	BBX4(N)	2	53.98
	BBX7(N)	3	22.91
	BBX8(N)	4	70.22
	BBX9(N)	5	75.57
	CKIMB(N)	4	35.16
	FK(N)	30	851.73
	WBBX2(N)	1	1.69
	WBBX5(N)	2	2.36
	WBBX9(N)	5	35.31
	WK(N)	16	293.61
	WKBBX4(N)	4	44.71
Centre	BBX1(C)	8	223.41
	BBX3(C)	6	104.66
	BBX4(C)	1	9.68
	BBX6(C)	5	27.87
	BBX9(C)	4	64.65
	CFK(C)1	22	986.15
	CFK(C)2	2	15.42
	CFK(C)3	1	38.49
	CKIMB(C)	7	45.57
	FK(C)	63	1132.86
	KBBX1(C)	3	29.29
	KBBX2(C)	2	41.98
	KBBX3(C)	7	36.45
	WK(C)	14	690.24
South	BBX(S)	7	46.57
	CBBX(S)	1	0.8
	CKBBX(S)	2	17.5
	CKIMB(S)	20	135.01
	E/MPK(S)	36	3410.26
	IntSWBas	5	69.05
	M/PK(S)	53	7506.32
	WBBX(S)	11	127.39
	WK(S)	31	1437.25
	WKBBX(S)	9	272.75
	WM/PK(S)	4	294.08

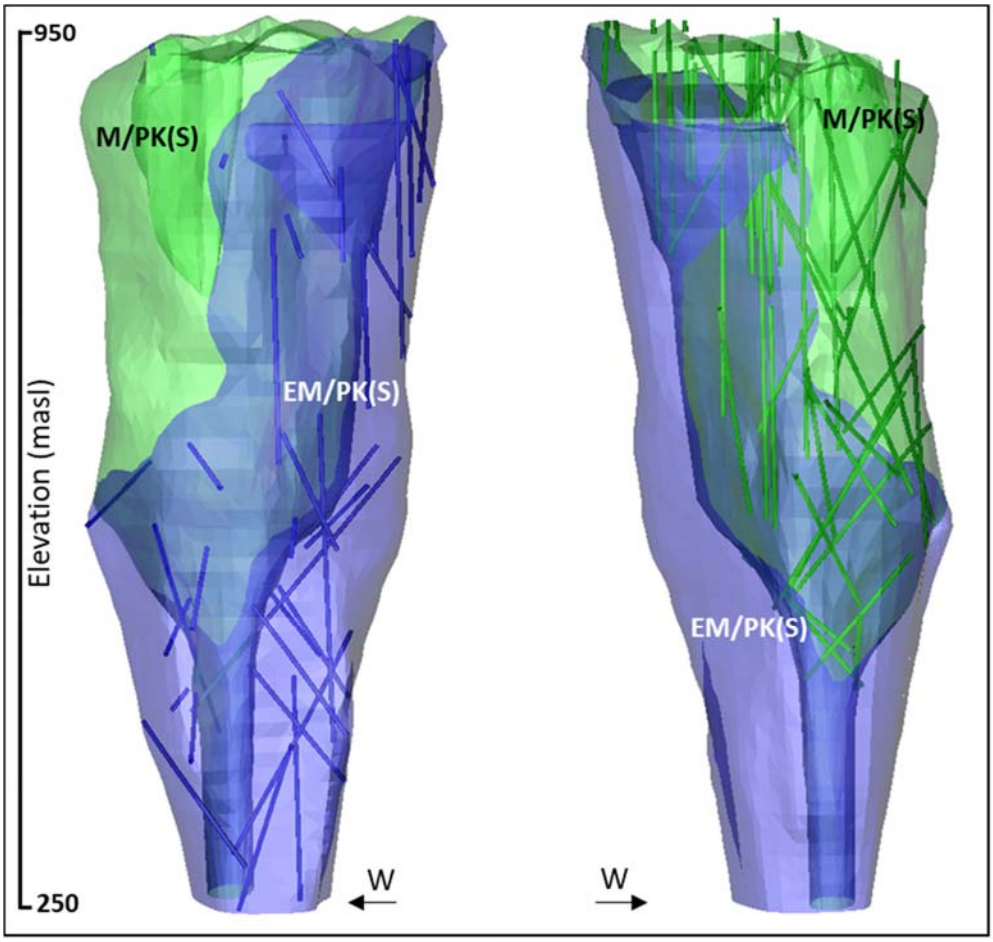

Figure 7-6: Inclined view oriented towards the north (left) and the south (right) illustrating the morphologies of the M/PK(S) and EM/PK(S) domains (both shown as transparent) and the internal core drill coverage that was used to define them (EM/PK(S) in blue and M/PK(S) in green).

8. Deposit types and mineralization characteristics

The primary source rocks for diamonds that are presently being mined worldwide are kimberlites, orangeites and lamproites. All of these are varieties of ultramafic (i.e. Fe and Mg-rich, Si-poor) volcanic and subvolcanic rocks defined by different characteristic sets of minerals. Of these rocks, kimberlites represent the vast majority of primary diamond deposits that are currently being mined.

Kimberlites are mantle-derived, volatile-rich (H_2O and CO_2) ultramafic magmas that transport diamonds together with fragments of mantle rocks from which the diamonds are directly derived (primarily peridotite and eclogite) to the earth's surface from great depths (>150 km depth). They are considered to be hybrid magmas comprising a mixture of incompatible-element enriched melt (probably of carbonatitic composition) and ultramafic material from the lower lithosphere that is incorporated and partly assimilated into the magma.

Coherent (previously termed magmatic) kimberlites are the products of direct crystallization of kimberlite magmas, and typically comprise olivine set in a fine-grained crystalline groundmass made up of 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. While some olivine crystallizes directly from the kimberlite magma on emplacement (to form phenocrysts), kimberlites generally include a significant mantle-derived (xenocrystic) olivine component that typically manifests as large (>1 mm) anhedral crystals. In addition to mantle-derived olivine, kimberlites also commonly contain other mantle-derived minerals, the most common and important being garnet, chrome-diopside, chromite and ilmenite. These minerals, 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.

The style of emplacement of kimberlite at or just below the surface of the crust is influenced by many factors which include the following:

- characteristics of the magma (volatile content, viscosity, crystal content, volume of magma, temperature, etc.);
- nature of the host rocks (i.e. unconsolidated mud vs. hard granite);
- local structural setting;
- local and regional stress field; and
- presence of water.

Kimberlites occur at surface as either sheet-like intrusions (dykes or sills) or irregular shaped intrusions and volcanic pipes. The sheets and irregular intrusions are typically emplaced along pre-existing planes of weakness in the country rock. Their emplacement does not involve explosive volcanic activity, and thus they are generally comprised of texturally-unmodified coherent kimberlite. In contrast, the pipes are generated by explosive volcanic activity related to the degassing of magma, or the interaction of magma and water, or a combination of both of these processes. This explosive volcanic activity typically produces

pieces or clasts of the kimberlite magma (and all the enclosed rock and mineral grains and fragments therein), as well as pieces of the country rock in which it was emplaced. Deposits derived directly or indirectly from volcanic processes which texturally-modify the primary components of kimberlite magma are termed volcaniclastic kimberlite.

Due to the wide range of settings for kimberlite emplacement, as well as varying properties of the kimberlite magma itself (most notably volatile content), kimberlite volcanoes can take a wide range of forms and be infilled by a variety of deposit types. This range is illustrated schematically in Figure 8-1. Volcanic kimberlite bodies range in shape from steep-sided, carrot-shaped pipes (diatremes) to flared champagne-glass or even “pancake” like crater structures. While diatremes are often interpreted to be overlain by a flared crater zone, there are few instances where both diatreme and crater zones are preserved (e.g. Orapa kimberlite in Botswana; Fox kimberlite at Ekati). Kimberlite volcanoes are infilled by a very wide range of volcaniclastic kimberlite types, ranging from massive, minimally-modified (texturally) pyroclastic kimberlite, to highly modified pyroclastic and resedimented volcaniclastic deposits that have been variably affected by dilution, fragmentation, sorting, and elutriation (removal of fines).

Diamonds are xenocrysts within kimberlite as they are primarily formed and preserved in the deep lithospheric mantle (depths > ~150 km), generally hundreds of millions to billions of years before the emplacement of their kimberlite hosts. The diamonds are “sampled” by the kimberlite magma and transported to surface together with the other mantle-derived minerals described above.

In general, diamonds can vary significantly within and between different kimberlite deposits in terms of total concentration (commonly expressed as carats per tonne or carats per hundred tonnes), particle size distribution and physical characteristics (e.g. colour, shape, clarity and surface features). The value of each diamond, and hence the overall average value of any given diamond population, is governed by the size and physical characteristics of the stones.

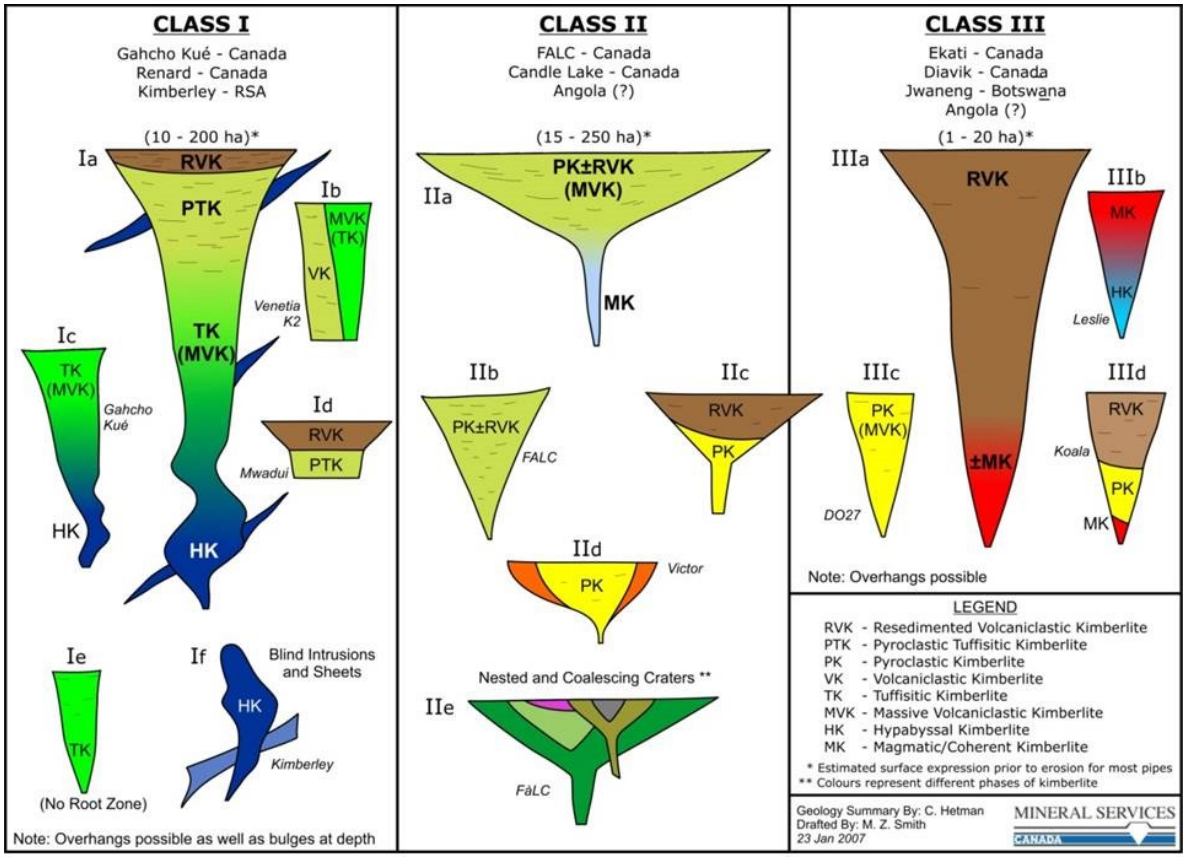

Figure 8-1: Schematic illustration of common shapes for kimberlite volcanic bodies based on observations from around the world. The three classes (I, II and III) represent broad groupings with shared attributes of geometry, size and infill.

The overall concentration of diamonds in a particular kimberlite deposit is dependent on several factors, including:

- the extent to which the source magma has interacted with and sampled potentially diamondiferous deep lithospheric mantle;
- the diamond content of that mantle (diamonds are only present locally and under specific pressure temperature conditions in the mantle);
- the extent of resorption of diamond by the kimberlite magma during it ascent to surface and prior to solidification;
- physical sorting and/or winnowing processes occurring during volcanic eruption and deposition; and
- dilution of the kimberlite with barren country rock material or surface sediment.

The diamond size distribution characteristics of a kimberlite deposit are inherited from the original population of diamonds sampled from the mantle but can be affected by a number of secondary

processes, including resorption during magma ascent and sorting during eruption and deposition of volcaniclastic kimberlite deposits.

The physical characteristics of the diamonds in a kimberlite deposit are largely inherited from the primary characteristics of the diamonds in their original mantle source rocks but can be affected by processes associated with kimberlite emplacement. Most notable of these are:

- chemical dissolution (resorption) by the kimberlite magma resulting in features ranging from minor etching to complete dissolution of the diamonds;
- formation of late stage coats of fibrous diamond either immediately prior to or at the early stages of kimberlite emplacement;
- physical breakage of the diamonds during turbulent and in some cases explosive emplacement processes.

9. Exploration

This section summarizes advanced exploration work (used to support resource estimates) on the AK6 kimberlite carried out 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 briefly summarised here (extracted and summarised from Oberholzer et al., 2017) and are detailed in Lynn et al., 2014, McGeorge et al., 2010 and various references therein. Recent exploration completed in 2017 included core drilling and sampling of core material and is documented in Sections 10.2 and 10.3. The current resource estimate is based on data collected during these programs incorporating results from mining operations and diamond sales since 2012 (Lynn et al., 2014; Oberholzer et al., 2017, this report).

The AK6 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 historical sampling, limited and shallow, 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 (Table 9-1) followed a staged approach, which can be summarized as follows:

- Early evaluation – 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 cpht 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 considered to be the limits of possible open pit mining based on an initial economic assessment.

- In 2016/17 two core drilling programs were conducted on the AK6 kimberlite. The combined drilled metres of 12,272 provided additional pierce points and geological information for the deeper portion of the South Lobe.

Table 9-1: Summary of major exploration phases at AK6.

Stage	Work done	Duration
Early evaluation	5 x 12¼" large diameter drill holes totaling 679 m, 97 tonne bulk sample. DMS and diamond recovery Geophysical surveys	2003 - 2005
Phase 1 advanced exploration	44 x 6½" percussion holes for delineation totaling 4,575 m 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 and diamond recovery from 1,775 tonnes	2005 - 2006
Phase 2 advanced exploration	11 x cored boreholes (NQ) as LDD pilots totaling 4,181 m 29 x inclined boreholes (NQ) for delineation totaling 8,679 m 12 x 23" LDD totaling 4,265 m Trench bulk sampling at surface DMS processing and diamond recovery from 2,235 tonnes	2006 - 2008
Delineation and geotechnical drilling	15 x cored borehole (HQ and NQ) totalling 12,272 m 916 microdiamond samples (7,315 kg)	2016 - 2017

9.2 Geophysical surveys

The AK6 kimberlite was first identified from an aeromagnetic survey in 1969. During 2005 De Beers implemented four high resolution ground geophysical surveys as outlined in Table 9-2. The geophysical data were used to support the development of the first AK6 geological model.

Table 9-2: High resolution geophysical surveys carried out over AK6.

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 low due to the weathering of the pipe.
Electromagnetics (Geonics EM34 frequency domain)	57.6	Approximately defined kimberlite contacts.
Controlled Source Audio-frequency Magneto-Tellurics (CSAMT)		Detected the three lobes at depth.

10. Drilling

10.1 Historical delineation and bulk sample drilling

Early drilling (2003 to 2007) of the AK6 kimberlite is described in detail in a previous Technical Report dated 25th March 2010 (McGeorge et al., 2010) and the references therein. A brief summary is provided here, extracted from Oberholzer et al. (2017). Drilling can be assigned to three main categories: (1) core drilling to delineate the extent of the kimberlite and to map its internal geology/density; (2) large diameter drilling (LDD) to obtain large kimberlite samples to support estimates of diamond grade and value; and (3) pilot core drilling adjacent to LDD holes confirm the geology and kimberlite units sampled. Drilling is summarized in Table 10-1, grouped into the exploration phases described in Section 9 above. Drill hole locations are illustrated in Figure 10-1.

Table 10-1: Historical (2003 to 2007) drilling at AK6.

Phase	Purpose	Drill type	Diameter	Holes	Metres	Period
Early evaluation	Bulk sampling	RC	12¼"	5	679	2003 - 2004
Phase 1 advanced exploration	Delineation	Percussion	6½"	44	4,575	2004 - 2005
	Delineation	Core	NQ	17	6,904	2005
	Piloting	Core	NQ	12	2,979	2005
	Bulk sampling	LDD	23"	13	3,699	2005 - 2006
Phase 2 advanced exploration	Piloting	Core	NQ	11	4,181	2005 - 2006
	Delineation	Core	NQ	29	8,679	2006 - 2007
	Bulk sampling	LDD	23"	12	4,265	2006 - 2008

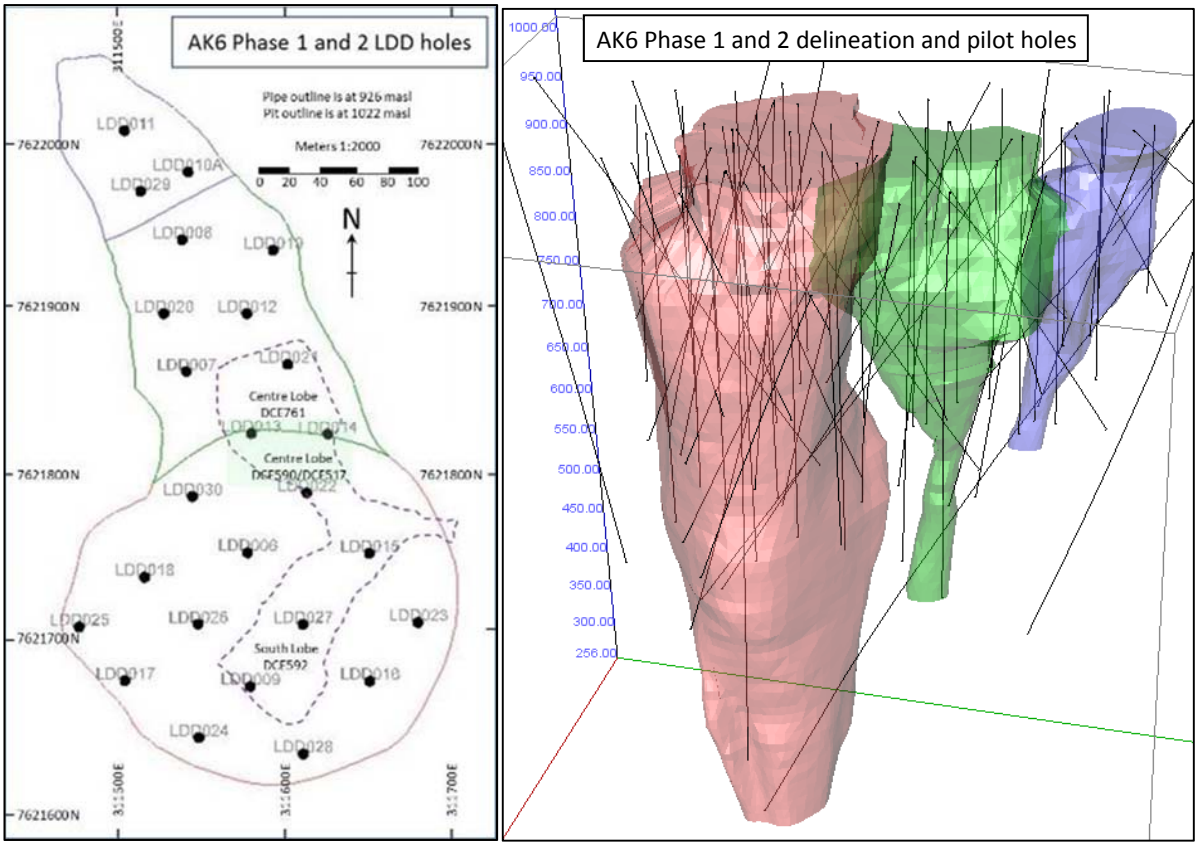

Figure 10-1: AK6 Phase 1 and 2 drill holes (see Table 10-1). Early evaluation holes are not shown as they were not used to support Mineral Resource Estimates. Large diameter RC holes (left, plan view) are all vertical, the outline of a surface trench bulk sample is shown as a dotted black line. Core drill holes (right, inclined view oriented towards the southwest) are shown as thin black traces with the South, Centre and North Lobes shown as red, green and blue, respectively.

10.2 Recent delineation and geotechnical drilling

Two drill programs were completed in 2017 to support further evaluation of the deeper portion of the South Lobe between 400 and 600 masl and to provide geotechnical information on host rock stratigraphy and physical properties. A total of 12,272 m were completed from 15 drill holes, as summarised in Table 10-2. Drill coverage is shown in Figure 10-2. For certain holes survey of azimuth and dip could not be completed (5 holes) to the base of the hole due to hole collapse and compression. Survey of azimuth and dip also produced highly irregular results in 2 holes. These drill holes with unreliable survey data were not used to support geological modelling, as discussed further in Section 12.1.1.

Table 10-2: Recent (2017) delineation (REP) and geotechnical (GT) drilling.

Drill hole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip	Comment
REP_001	341111	7621702	1,014	854	94	-49	
REP_002	341579	7622200	1,011	801	189	-46	Survey incomplete
REP_003	341553	7621337	1,014	807	353	-55	
REP_004	341064	7621744	1,014	893	92	-50	
REP_005	341629	7622168	1,012	758	201	-40	
REP_006B	341270	7622221	1,012	917	156	-44	
REP_007	341939	7621891	1,012	818	246	-54	Survey incomplete
REP_008	341236	7621748	1,013	755	88	-57	Survey incomplete
REP_009	341074	7621740	1,014	918	101	-55	Survey incomplete
REP_010	341937	7621891	1,012	809	245	-51	Not surveyed
REP_011	341230	7621751	1,013	668	112	-48	
REP_012	341942	7621880	1,012	753	249	-49	Survey unreliable
GT01a	341319	7621476	1,013	742	44	-55	Survey unreliable
GT02a	341777	7622090	1,012	902	207	-55	
GT03	341916	7621503	1,013	875	298	-61	
			Total	**12,272**			

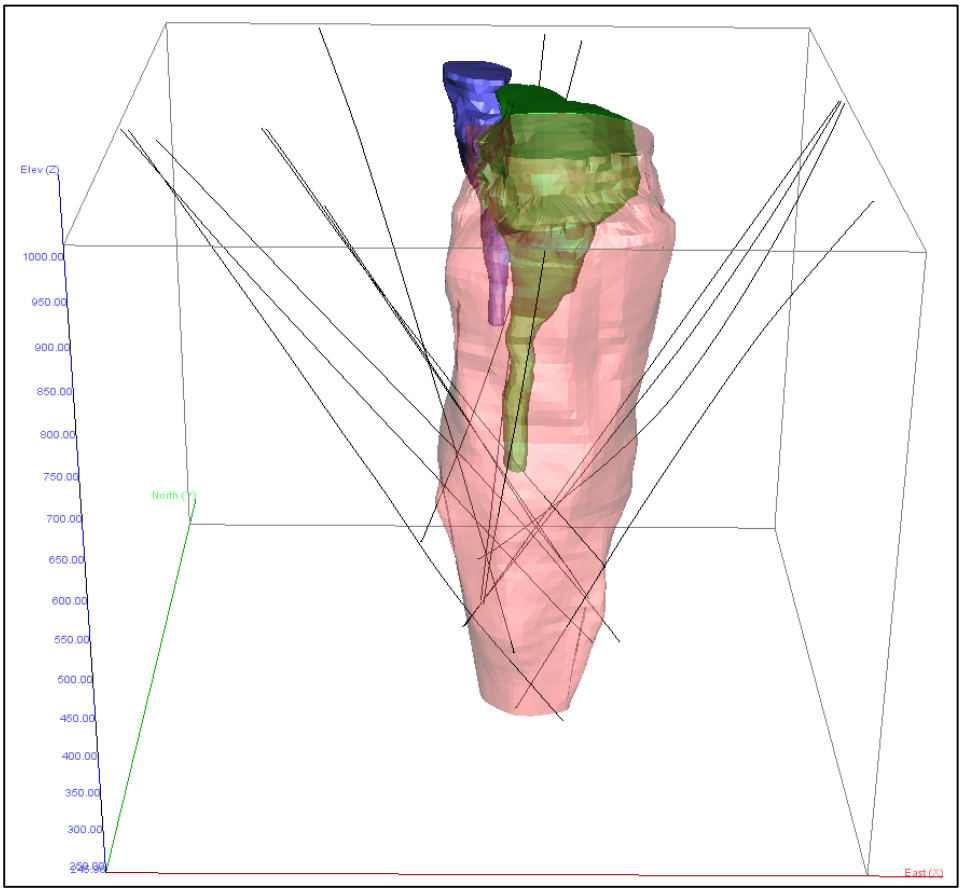

Figure 10-2: Inclined view, oriented towards the north, showing AK6 recent (2017) delineation and geotechnical drill holes. The South, Centre and North Lobes shown as red (transparent), green and blue, respectively.

10.3 Drill core sampling

Sampling of drill material in support of historical resource estimates has been well documented in previous Technical Reports (Lynn et al., 2014 and McGeorge et al., 2010). This section provides details on previously unreported sampling work carried out in 2017 on recent (Section 10.2) and historical (Section 10.1) cores in support of this updated Mineral Resource Estimate. A fundamental aspect of this estimate has been the demonstration of geological continuity within the M/PK(S) and EM/PK(S) units with depth (Sections 14.2.2, 14.4.3 and 14.4.4). Sampling from historical drill cores was necessary to represent the shallower portions of the South Lobe, as recent (2017) drilling focussed on the deeper area between 600 and 400 masl. Sample coverages achieved are shown in Figure 10-3. Sampling was undertaken for bulk density, petrography and microdiamond[1] analysis, as follows:

- Bulk density samples (n = 342). Samples each comprised approximately 10 to 20 cm of whole core collected from recent (2017) drill core only; historical drill cores were comprehensively sampled for bulk density.
- Petrography samples (n = 227) were collected from 13 of the 15 deep REP/GT drill cores (135 samples from below 600 masl) and from 10 historical drill cores (92 samples providing broad coverage of the M/PK(S) and EM/PK(S) units above 600 masl). The samples were collected at regular 20 m intervals in the REP/GT holes and at 10 to 30 m intervals in historical holes. Each sample comprised 15 to 25 cm of whole core.
- Microdiamond samples (n = 916) were collected from 12 of the 15 deep REP/GT drill cores (total 470 samples) to achieve a broad spatially representative sample of the South Lobe below 600 masl and from 9 historical pilot drill cores (total 446 samples) adjacent to LDD holes to support investigations of the relationship between microdiamonds and macrodiamonds[2] in the M/PK(S) and EM/PK(S) units (see Section 14.4.6). Samples comprised whole core of lengths varying between approximately 1 and 2 m, depending on core diameter (samples were collected to achieve an 8 kg mass to meet laboratory processing constraints). Sample spacing varied from approximately 5 m to continuous depending on the expected grade of the material and the objectives of the sampling (see Section 14.4.2).

[1] The term microdiamond is used throughout this report to refer to diamonds recovered through caustic fusion of kimberlite at a bottom screen size cut-off of 0.105 mm (~0.00002 ct). Rare larger diamonds that may be recovered by a commercial production plant may be recovered through this process but are still referred to as microdiamonds.
[2] The term macrodiamond is used throughout this report to refer to diamonds recovered by diamond production plants, which typically only recover diamonds in and larger than the Diamond Trading Company (DTC) sieve category 1 (i.e. > ~0.01 ct).

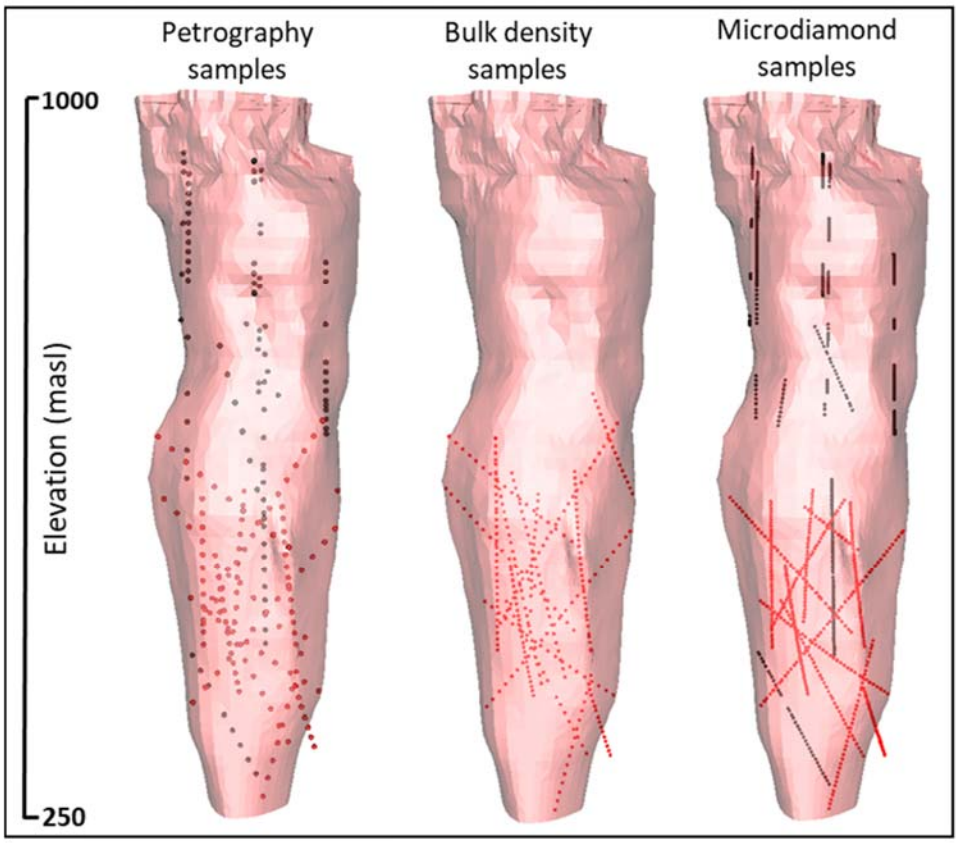

Figure 10-3: Inclined view, oriented towards the east, showing locations of samples collected from drill core in the South Lobe in 2017 in support of this updated Mineral Resource Estimate. Samples are coloured red or black if they were collected from recent or historical drill cores, respectively (see text above).

11. Sample preparation and analyses

The sample preparation, analyses and security measures applied to samples from the original evaluation programs (by De Beers during the period 2003 to 2007) are described in the previous Technical Reports (McGeorge et al., 2010 and Lynn et al., 2014) and are provided here (Section 11.1, extracted and summarized from Oberholzer et al., 2017) for reference. Previously unreported information relating to samples collected during 2017 (see Section 10.3) in support of this updated Mineral Resource Estimate is provided in Sections 11.2 to 11.4.

11.1 Historical samples

11.1.1 LDD reverse flood, 23" drill samples

These samples were collected during Phase 1 and 2 exploration (Section 9.1) from LDD holes described in Section 10.1. They form the basis of the grade estimate above 604 masl described in Section 14.4.5.

Sample material recovered from drilling 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 the supervision of a geologist. It was then transported to the DMS plant at the De Beers Letlhakane camp by truck, also under the charge of the geologist. At the camp, the responsibility for the sample passed to the plant foreman. The processing plant was a 10 tonnes per hour mobile DMS unit. A total of 4,010 t of +1 mm sample were processed, yielding 306 t of concentrate. The Central and North Lobe concentrate yields averaged 1.1 %, while yields from the South Lobe were higher, with averages of between 6 and 8 %.

Following DMS 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 transported 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.

Diamond recovery was carried out at GEMDL in Johannesburg. The diamond recovery parameters at GEMDL were the same for all phases. The GEMDL facility was fully ISO17025 certified at the time of sample processing. 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. A de-falsification process was carried out to remove mis-identified material; where necessary an infra-red spectrometer was used to confirm diamond.

All equipment and floors were purged between consignments. For quality assurance, tracer diamonds were added to the sample by an external monitoring team. After de-falsification, the monitor diamonds were removed. The diamonds were then sent to Harry Oppenheimer House in Kimberley, South Africa, for acid cleaning, re-sieving and final weighing to record stone counts and carat weights per DTC sieve size class. The X-ray tailings were reconstituted and put into 50 litre blue plastic drums, packed into 6 m shipping containers, and returned to site.

11.1.2 Bulk density samples

Bulk density measurements were carried out on core samples using a water immersion method, by taking a 15 cm length of core and weighing it in air and in water, drying the sample prior to re-weighing 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.

11.1.3 Microdiamond samples

The historical microdiamond dataset for AK6 (77 samples, 1,436 kg) derives from both core and reverse circulation drill chip material. The methods by which these samples were processed and microdiamonds recovered are not known and the results are not considered reliable (Section 12.5).

11.2 Petrography samples

All petrography samples collected in 2017 were labelled with the drill hole number, depth and way-up direction by Boteti geologists. No further sample preparation was carried out on site and petrography samples were shipped to Vancouver Petrographics Ltd. for processing under the “dry” petrographic sample preparation method. A polished slab preserved with epoxy and two thin sections (standard and wedged) were produced for each sample, for examination under Nikon binocular and petrographic microscopes. Polished slabs, off-cuts and thin sections are in storage at the MSC offices in Vancouver, Canada.

11.3 Bulk density samples

All bulk density sample processing in 2017 was carried out on site by Boteti geologists. Sample masses were recorded at an on-site laboratory and sample volumes were determined by a water-immersion method as per Lipton (2001). No drying of samples was carried out; the bulk density measurements collected in 2017 are not of dry bulk density, and a minor adjustment to account for moisture content (and ensure compatibility between the new and historical datasets) was carried out as documented in Section 12.3.

11.4 Microdiamond samples

No preparation of microdiamond samples collected in 2017 was carried out on site. Samples of whole core were collected, securely bagged and packaged into 20 l drums for shipping to the Saskatchewan Research Council (SRC) Geoanalytical Laboratory in Saskatoon, Canada. Sample drums were sealed with security tags prior to shipping and the tags were verified by SRC upon receipt. Processing information in this section was provided by the SRC and their process flowsheet is shown in Figure 11-1.

Each 8 kg sample is loaded into a 40 l furnace pot with 75 kg of virgin caustic soda (NaOH). Bright yellow synthetic diamonds between 0.15 and 2.12 mm in size are added to alternating samples as QA/QC spikes. The furnace pot is heated in a kiln to 550°C for 40 hours and then removed and allowed to cool. The molten sample is poured through a 0.106 mm screen, which is then discarded after use. Micro-diamonds and other insoluble minerals (typically ilmenite and chromite) remain on the screen. The furnace pot is then soaked with water to remove any remaining caustic and microdiamonds. The water is poured through the same screen. Samples are then acidized to neutralize the caustic solution. The residue is then rinsed and treated with acid to dissolve readily soluble materials. Samples are then transferred to a zirconium crucible along with yellow synthetic diamonds spikes (to alternating samples not spiked prior to fusion) and fused with sodium peroxide to remove any remaining minerals other than diamond from the sample. The sample is allowed to cool and is then decanted through wet screens to size diamonds according to Canadian Institute of Mining and Metallurgy (CIM) square mesh sieve classes. All diamonds are counted and weighed. Individual stone descriptions for all diamonds larger than 0.3 mm are recorded. Stones are stored in plastic vials filled with methanol.

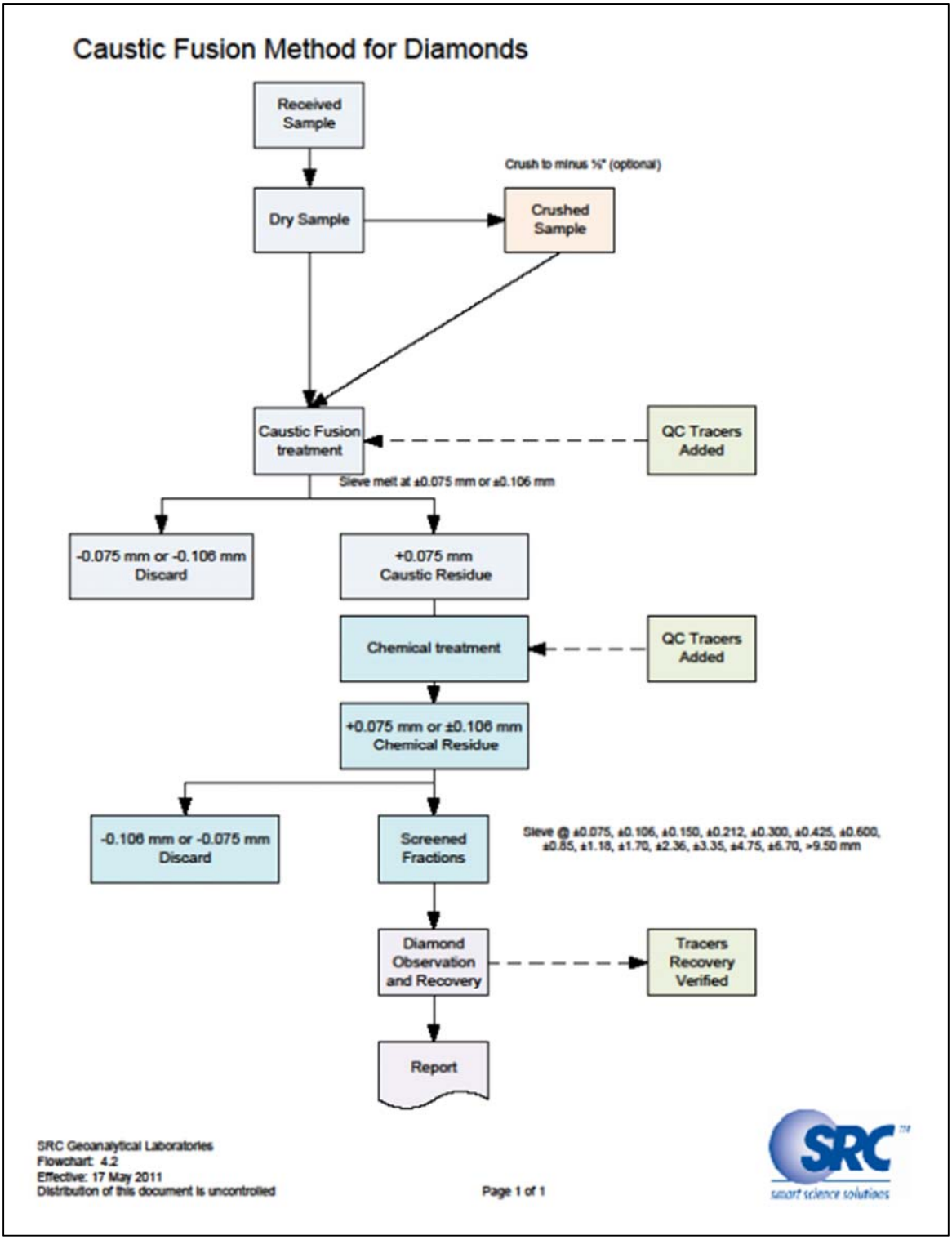

Figure 11-1: Processing flowsheet for microdiamond samples processed at the Saskatchewan Research Council.

12. Data verification

12.1 Geological model

12.1.1 Drill hole and orientation surveys

During the original evaluation of AK6 (2003 to 2007) all drill hole core hole collar positions were surveyed with a Leica DGPS 500 system. Core hole orientation surveys were carried out with either magnetic- or gyroscope-based survey systems, and the magnetic-based surveys were considered to be of low confidence (McGeorge et al., 2010). The results were reviewed and the geological model produced on the basis of the early drilling results was considered to be of sufficiently high confidence to support estimation of Mineral Resources (Lynn et al., 2014).

During recent (2017) drilling there were significant issues with downhole orientation surveys. Due to the length and shallow inclination of the holes it was frequently not possible to survey the entire length of each hole due to hole compression and collapse. All holes were surveyed with a gyroscope-based tool and while the dips recorded are considered accurate there are instances in which the recorded orientation data show unrealistic deviations. A complete review of all survey data from 2017 drilling was carried out (MSC17/006R) and 11 of 31 new pierce points were discarded as unreliable.

12.1.2 Mine survey data

The geological shell model has been updated in 2014 and in 2018 on the basis of mine survey records of the pipe contact where exposed at surface. 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 (Oberholzer et al., 2017).

12.1.3 Geological logs and internal geology

The original AK6 internal geological model (McGeorge et al., 2010) was developed on the basis of drill core logs, petrography work and whole rock geochemistry (trace element ratios). The integrated results of these were used to identify kimberlite units that were modelled as discrete domains to support resource estimation. The data and methods supporting this work have been comprehensively audited during compilation of the various project Technical Reports and a summary of this audit work was presented in Oberholzer et al. (2017). Mining carried out since 2012 has confirmed only minor inaccuracies in the internal geology (previously unrecognised high-dilution breccia and internal basalt raft) and the shell model, and updates were incorporated into the geological model reported subsequently (Lynn et al., 2014).

Recent (2017) drilling confirmed that the EM/PK(S) unit is in fact the volumetrically dominant unit below ~550 masl in the South Lobe and core logging / extensive petrography work on new drill core allowed for a more robust categorisation of the characteristic features of EM/PK(S) in comparison with M/PK(S). This in turn supported a review exercise of the South Lobe drill core photo records in which previously mis-

identified EM/PK(S) in historical drill core was correctly re-logged, as documented in MS18/005R. This comprehensive review resulted only in the remodelling of the internal boundary between the EM/PK(S) and the M/PK(S) domains below the weathered horizon; no changes (other than updates for new pipe margin exposures) were made to the model above the weathering horizon (~910 to 930 masl).

12.2 Internal dilution data

Estimates of the volume percent of wall-rock fragments (internal dilution) exceeding 0.5 cm in size were determined for historical (2003 to 2007) drill core 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. Measurement of wall-rock fragments exceeding 0.5 cm in size were determined for recent (2017) drill core by line scan over approximate 1 m intervals for all drill core on a continuous basis down hole. The methods used are considered by MSC to be appropriate and consistent with industry best-practice and the results are considered to provide (1) reasonable constrains on the average internal dilution present in the domains and (2) reasonable constraints on the spatial variation in dilution within volumetrically significant domains that are well represented by data, particularly M/PK(S) and EM/PK(S). Independent analysis by MSC of the historical line scan data yielded dilution estimates that were not materially different to those obtained by De Beers.

12.3 Bulk density

The bulk density data used for estimation prior to 2018 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 measurements were done on core samples using a water immersion method consistent with Lipton (2001), by taking a 15 cm length of core and weighing it in air and in water, drying and reweighing to calculate moisture and 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 geology model solids, and further checked the data against original De Beers sample inventories for transcription errors. No significant data discrepancies were identified.

Additional bulk density data from 2017 drill core were generated using the same water immersion method, however samples were not dried subsequent to determination of volume and the measurements generated are not true “dry” bulk density. The new data were compared with the old data and no significant discrepancies were noted, due to the fact that the new bulk density data derive from deeper portions of the pipe (below 700 masl) where kimberlite is unweathered and characterised by very low (<2 %) moisture contents. Very minor adjustments for typical average moisture content in each domain by elevation range (as defined by measurements of moisture content for deep samples from historical holes, Table 12-1) were applied to the new data based on their elevation range to ensure consistency between datasets.

Table 12-1: Adjustments made to the entire 2017 bulk density dataset by domain and elevation range to account for moisture content (2017 samples were not dried during measurement of bulk density). Corrections are derived from historical bulk density samples in the corresponding elevation ranges.

Domain	Elevation range (masl)	Historical samples	Average moisture (%)
M/PK(S)	600 to 700	181	0.9
	400 to 600	51	0.6
EM/PK(S)	600 to 700	37	2.1
	200 to 600	86	0.8

12.4 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. Caliper surveys of down hole diameter were carried out to ensure the accuracy of the sample volumes used in grade calculations. The grade dataset 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 2018 and the grade estimate (average grade with depth) was found to be locally over-estimated where the grade interpolation was strongly influenced by several statistical outlier grade points. These outlier points were therefore subjected to a grade-capping exercise as documented in Section 14.4.1 prior to their use in grade estimation.

12.5 Microdiamond data

All microdiamond sample results used in this Mineral Resource Estimate were generated through processing at the Saskatchewan Research Council (SRC) in Saskatoon, Canada. The SRC employs a thorough system of quality control and the results generated are considered to be of high quality. The SRC adds synthetic diamonds to samples prior to fusion (Spike 1) and during chemical treatment of caustic residues (Spike 2) as outlined in Section 11.4. Recoveries of these synthetic diamonds are reported along with microdiamond results and were reviewed by MSC. Tracer losses are shown in Figure 12-1. No tracer loss was present in 809 of 916 samples processed. Spike 1 tracers were lost from 82 samples, Spike 2 tracers were lost from 25 samples. Maximums of 5 and 4 tracers were lost during Spike 1 and 2, respectively. The results imply sporadic occasional loss of diamond with no systematic issues likely to have compromised the results, which are considered to be of adequate quality for use in this Mineral Resource Estimate.

Processing methods for historical microdiamonds samples (Section 11.1.3) are not known. The historical dataset shows significant discrepancies with newly obtained data, and the historical results have therefore not been used.

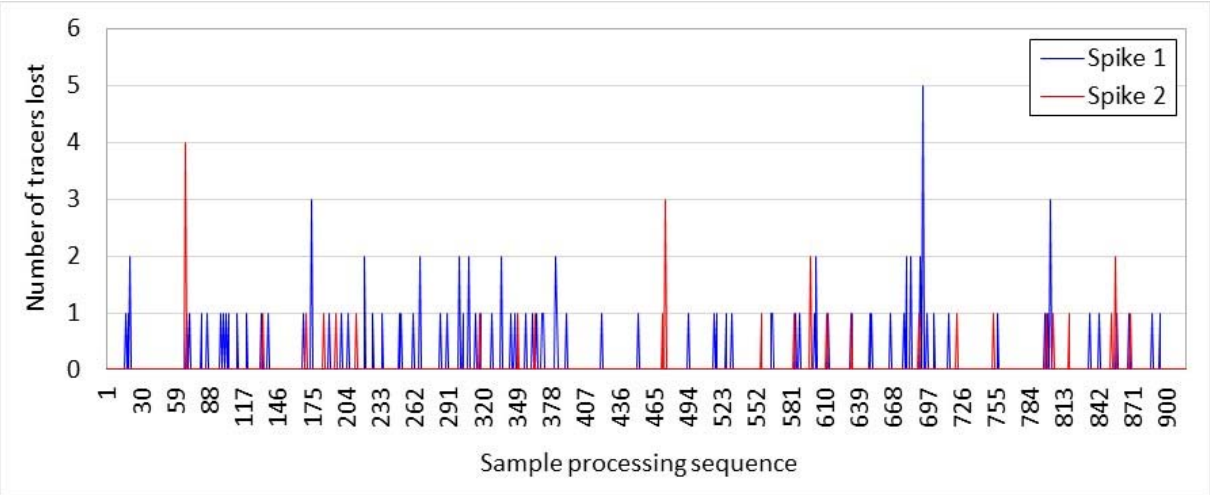

Figure 12-1: QA/QC spike recoveries from microdiamond samples. Spike 1 tracers were added prior to fusion. Spike 2 tracers were added during chemical treatment of caustic residues.

12.6 Production and sales data

12.6.1 Grade control data

The AK6 kimberlite has been mined for diamonds at Karowe Mine since April 2012. Detailed records of all kimberlite hauled are maintained by Boteti. 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). Since 2014 material derived from different lobes has been stockpiled separately. Related haulage and stockpile data are captured by Boteti 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 ore movement on site and can be used with a high level of confidence (since 2014) to confirm the source material for plant production where the material was moved directly from the pit to the plant. Prior to 2014 accurate records of stockpile material feed to the plant were maintained but kimberlite from different source locations was blended on the stockpiles. 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.

12.6.2 EM/PK(S) controlled production run

The remodelled EM/PK(S) domain (Section 7.4.2) is exposed at surface in the open pit and a controlled production run was carried out to process material derived from within this domain. The contact between M/PK(S) and EM/PK(S) was mapped where exposed and accessible (due to safety concerns near pit walls) by Boteti geologists. Magnetic susceptibility measurements were used to guide this process, as EM/PK(S) presents a distinctively lower susceptibility than M/PK(S) (Section 7.3.4). Mapping was reviewed by Dr Armstrong on site, who confirmed the extracted material was derived entirely from within the EM/PK(S) domain and further supervised the QA/QC protocols for ore stockpiling and processing. Ore was mined from a single location and stored as a single stockpile, and material was derived exclusively from this stockpile during processing. The front end of the plant (prior to mill feed) was inspected, all bins were drawn empty, spillage was cleaned up and the primary crusher bin was drawn empty. All X-Ray Transmission (XRT) storage bins (4-8 mm, middles, coarse and large diamond recovery circuits) were drawn empty, the Dense media separation (DMS) feed bin was emptied and recovery was purged. The primary stockpile was drawn down and pushed in, surveyed and then EM/PK(S) material was crushed and fed to the stockpile. EM/PK(S) was fed to the plant for 24 hours prior to the recorded commencement of the reported EM/PK(S) controlled production run (Section 14.4.1).

12.6.3 Ore processing and diamond recovery

In 2013 the Karowe Mine plant process was reviewed (Lynn et al., 2014) 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.

12.6.4 Diamond production 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. In 2014 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. Diamond data subsequent to 2014 have not been audited by MSC but have been recorded by the same methods / workflows and the data provided by Boteti / Lucara are considered reliable.

12.6.5 Sales data

Pre and post-sales reports for all Karowe diamond sales (including separate large stone tenders) since inception of production were provided to MSC. No comprehensive audit was carried out, but sales results

for selected lots were compared with the data compilations used to estimate diamond value and no discrepancies were noted.

13. Mineral processing and metallurgical test work

It was recognised during the Feasibility Study stage of the Karowe Mine that there were significant metallurgical risks in the ability of the grinding circuit to process hard kimberlite below the weathered zone and in 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. The recovery of exceptionally large, high value diamonds necessitated further assessments of the recovery circuit to limit diamond breakage. Communition test work, assessments of X-Ray Transmission (XRT) diamond recovery technologies and diamond breakage studies were commissioned in 2013 (Lynn et al., 2014) to investigate technologies to mitigate these risks. The Karowe plant was modified (May 2015) based on the results of these as shown in Figure 13-1. This major plant modification was referred to as “Phase 2” upgrades in Oberholzer et al. (2017).

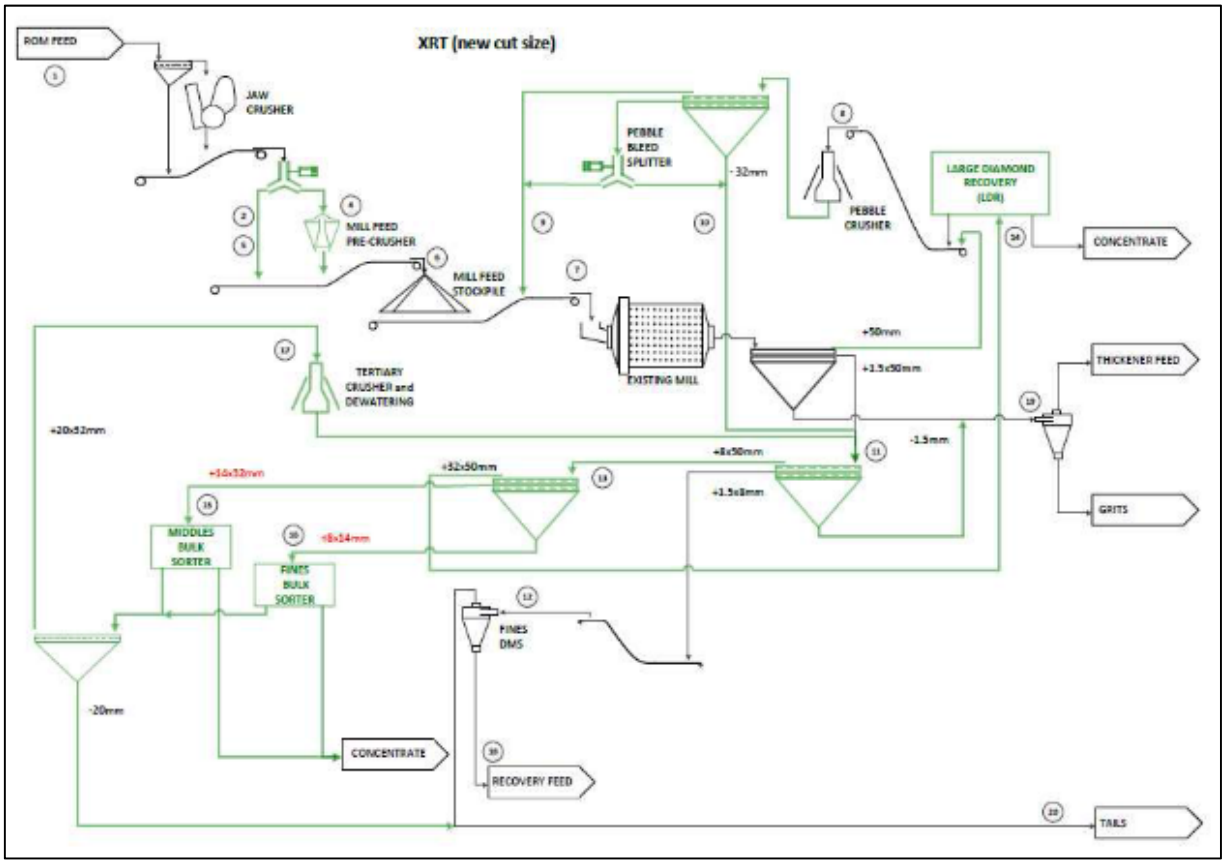

Figure 13-1: Flowsheet for the Karowe Mine process plant. The original plant process (as commissioned in 2012) is shown in black. Phase 2 modifications are shown in green. Figure extracted from Oberholzer et al. (2017).

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

14. Mineral Resource Estimate

Mineral Resources for AK6 have been previously reported in project Technical Reports and in a Preliminary Economic Assessment of underground mining potential in various stages as follows:

1. McGeorge et al. (2010) reported Mineral Resource Estimates based on the work of De Beers between 2003 and 2007 (see Sections 6, 9, 10 and 11), which culminated in a Mineral 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).
2. An updated Mineral Resource Estimate was reported by Lynn et al. (2014), in which minor modifications to the geological model were integrated with a major revision to average diamond value (based on diamond sales from production).
3. Further adjustments to the grade model (capping for grade outliers) and to the average value estimates were made in a Preliminary Economic Assessment of the Karowe Underground Project reported in Oberholzer et al. (2017).

This current report incorporates the results of recent core drilling (Section 10.2) and extensive drill core sampling (Section 10.3) completed in 2017 to present an updated Mineral Resource Estimate.

14.1 Approach to Mineral Resource Estimate

Mineral Resource Estimates for AK6 are based on a geological model (constraining the volume of the body and its internal domains) combined with estimates of bulk density, grade and diamond value.

The geological model for AK6 (Section 7.4) has been updated from that reported in Oberholzer et al. (2017) based on the results of recent core drilling and review of historical core logs. The near-surface pipe shell has also been modified slightly to reflect survey of its location as mapped from mine exposures.

A block modelling approach has been used for estimation of volumes, tonnes and grade for the AK6 kimberlite. To accommodate the numerous domains present, a partial (percent) block modelling approach was applied using a Dassault Systemes Geovia™ GEMS (GEMS) block model with the following parameters:

- Block model origin (X, Y, Z): 341000, 7621170, 1024 (coordinates defined in the Universal Transverse Mercator (UTM) coordinate system in the WGS84 datum for Zone 35S).
- The block model is not rotated in any direction, the Y axis points north.
- Block size 12 by 12 by 12 m. Note that the block size has been decreased from that previously used to report AK6 Mineral Resources (25 by 25 by 12 m, Oberholzer et al., 2017).
- Block model comprised of 104 columns, 102 rows and 65 levels, equating to a total of 689,520 blocks. Note that the overall block model extents have been reduced from those reported in Oberholzer et al. (2017) to accommodate the decrease in block size without creating an excessively large total number of blocks.

The block model folder structure was simplified from that used for previous estimates (Lynn et al., 2014, Oberholzer et al., 2017) to combine resource domains where possible (Table 14-1, Section 14.2), while still accommodating the grouping parameters required for grade and bulk density estimation (Sections 14.3 and 14.4). Resource domain model solids were integrated with the block model to capture the percentage of each domain within each block, calculated using the GEMS needling function with a horizontal needle orientation (10 x 10 needle density) and a minimum of 0.001 % volume required for the block to be populated. Domain volumes reporting from the block model were compared with the volumes of the 3D wire-frame solids and were found to be accurate to within 0.005 %.

Bulk density and grade estimates are based on two different approaches reflecting the evaluation data available:

- Above 604 masl (in all three lobes) local bulk density and grade estimates are based on interpolation of well distributed sample results (drill core bulk density samples and LDD grade samples) into a block model. The grade estimates generated by this approach are made on a per unit volume basis (cpm[3]) and reflect the efficiency with which diamonds were liberated and recovered from LDD samples at a +1 mm bottom cut off.
- Below 604 masl (in the South Lobe[1]) global average bulk density estimates by elevation range and geological domain were initialised directly into the block model. For this portion of the deposit average grade estimates were generated for each domain based on the results of a comprehensive microdiamond[2] sampling program with two components: 1) processing of large volumes of drill core from pilot holes adjacent to LDD holes to calibrate the ratio between microdiamond stone frequency (stones per kilogram) and LDD-recovered macrodiamond[3] grade (carats per tonne) for each of the two domains present below 604 masl, i.e. M/PK(S) and EM/PK(S); and 2) application of these ratios to domain average stone frequencies recovered from a large spatially representative microdiamond sample (below 604 masl) from each domain to derive average +1 mm LDD-recovered grade estimates on a per unit mass basis (cpt). These grade estimates were combined with the average bulk density estimates by elevation range to derive +1 mm carat per cubic metre grades that are directly comparable to the grade estimates made above 604 masl.

[1] The North Lobe does not extend below 604 masl. Only a very small volume of the Centre Lobe extends below 604 masl and the grade and bulk density of this material was therefore estimated using the same approach as above 604 masl. For the purpose of grade and bulk density estimation only the South Lobe is considered to extend below 604 masl.

[2] The term microdiamond is used throughout Section 14 to refer to diamonds recovered through caustic fusion of kimberlite at a bottom screen size cut-off of 0.105 mm (~0.00002 ct). Rare larger diamonds that may be recovered by a commercial production plant may be recovered through this process but are still referred to as microdiamonds.

[3] The term macrodiamond is used throughout Section 14 to refer to diamonds recovered by diamond production plants, which typically only recover diamonds in and larger than the Diamond Trading Company (DTC) sieve category 1 (i.e. > ~0.01 ct).

The +1 mm grade estimates made in this way were adjusted to reflect the differing efficiency with which the current Karowe Mine production plant, operating with a 1.25 mm bottom cut off, recovers diamonds in relation to the LDD sample process at a 1 mm bottom cut off. The block model was populated with these recovery-adjusted grade estimates to allow for extraction of the Mineral Resource Estimate through volumetric reporting.

Average diamond values are based on sales representing almost 6 years of production and are made on the basis of a well-constrained value distribution model (diamond value per sieve size class) for AK6 diamonds combined with diamond SFD models (percentage carats in each sieve size class) for each lobe as constrained by production (i.e. reflecting the recovery efficiency of the Karowe production plant at a bottom cut-off of 1.25 mm).

Details of the data and methods used to generate each component of the AK6 Mineral Resource Estimate are provided in the sections below.

14.2 Resource volumes

14.2.1 Resource domains and volumes

The geological model domains described in Section 7.4.2 have been adopted as the resource domains for this Mineral Resource Estimate. A summary of the domain names and total volumes is provided in Table 14-1. This table provides further information on block model codes and groupings that were used for bulk density and grade estimation, as discussed in Sections 14.3 and 14.4.

Table 14-1: AK6 resource domain volumes and interpolation groupings for grade and bulk density estimates. Volumes do not account for mining depletion and were extracted through volumetric reporting from GEMS. Rock codes are the unique numbers used to identify resource domains within the block model. Mm^3 = million cubic metres.

Lobe	Resource domain	Volume (Mm^3)	Description	Rock code	Bulk density group	Grade group
South	INTSWBAS	47	SW internal basalt raft	101	Breccia	N/a
	CKIMB(S)	142	South weathered	102	South weathered	South primary
	WK(S)	1,855				
	WBBX(S)	256	South weathered breccia	103		South breccia
	WKBBX(S)	196				
	BBX(S)	89	South breccia	104	Breccia	
	CBBX(S)	11				
	CKBBX(S)	27				
	KBBX(S)	0.3				
	M/PK(S)	11,953	South M/PK(S)	105	South primary	South primary
	WM/PK(S)	188				
	EM/PK(S)	7,467	South EM/PK(S)	106		
	South Lobe	**22,230**				
Centre	CKIMB(C)	80	Central/North weathered	201	Central/North weathered	Central/North primary
	WK(C)	829				
	WBBX5(C)	8	Central/North weathered breccia	202		Central/North breccia
	BBX1(C)	126	Central/North breccia	203	Breccia	
	BBX2(C)	10				
	BBX3(C)	67				
	BBX4(C)	1				
	BBX5(C)	11				
	BBX6(C)	19				
	BBX9(C)	42				
	KBBX1(C)	23				
	KBBX2(C)	28				
	KBBX3(C)	23				
	CFK(C)1	769	Central/North kimberlite	204	Central/North primary	Central/North primary
	CFK(C)2	2				
	CFK(C)3	9				
	FK(C)	1,497				
	Centre Lobe	**3,541**				
North	CKIMB(N)	53	Central/North weathered	301	Central/North weathered	Central/North primary
	WK(N)	300				
	WBBX2(N)	2	Central/North weathered breccia	302		Central/North breccia
	WBBX5(N)	14				
	WBBX9(N)	12				
	WKBBX4(N)	26				
	BBX1(N)	175	Central/North breccia	303	Breccia	
	BBX2(N)	24				
	BBX4(N)	12				
	BBX5(N)	1				
	BBX7(N)	7				
	BBX8(N)	25				
	BBX9(N)	41				
	KBBX1(N)	2				
	KBBX4(N)	1				
	FK(N)	426	Central/North kimberlite	304	Central/North primary	Central/North primary
	North Lobe	**1,119**				
Total Volume		**26,890**				

14.2.2 Geological continuity

Demonstration of geological continuity within each of the main kimberlite units is a key requirement for certain aspects of the Mineral Resource Estimate, in particular (1) the assignment of average diamond values (derived from near-surface production data) to kimberlite at depth and (2) the assignment of average grade estimates below 604 masl. Historical AK6 geology reports do not indicate any major geological discontinuity with depth within the volumetrically dominant kimberlite units, and grade variations within the units appear to be largely due to locally variable amounts of country rock dilution (Stiefenhofer, 2007; Stiefenhofer and Hanekom, 2005). To assess the degree of geological continuity MSC reviewed surface exposure, drill core and dilution measurements, and implemented a large petrographic study. This work has confirmed that, with the exception of local variations in the amount of country rock dilution for the FK(C) and FK(N) units, the key kimberlite units identified at AK6 are internally homogeneous with depth. The key findings from these assessments are described below.

Surface and drill core observations

Kimberlite exposures in the open pit were examined by MSC staff during site visits in July 2013, October 2013 and June 2017. Drill cores were briefly examined during site visits in July and October 2013, and detailed review of 10 complete drill cores was undertaken on site during June 2017. A complete photo review of all 2017 drill cores and of South Lobe historical core photographs was carried out in support of the 2018 update to the geological model, as documented in Section 7.3. 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 within the defined kimberlite units or between these units and their weathered equivalents. The main kimberlite units 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 were collected during historical and recent (2017) core drilling. Historical measurements (n = 3,377) were collected over 1 m intervals at an approximate spacing of 5 m down hole. Recent dilution measurements (1,466) were collected over approximate 3 m intervals on a continuous basis down hole. The datasets are therefore not directly comparable but measurements have been collected using the same method (recording the percentage dilution larger than 0.5 cm) and have been integrated as they do provide a reliable broad-scale assessment of the dilution characteristics of the major kimberlite units. The results suggest minor local variation and no significant large-scale dilution trends with depth in the main kimberlite units in the South Lobe (Figure 14-1). The amount of dilution present in FK(C) and in FK(N) is on average approximately double that of the South Lobe M/PK(S) and EM/PK(S) units and is more variably distributed. Potential grade variation associated with variation in dilution in the FK(N) and FK(C) units is accounted for in the local grade interpolation method used for these units (Section 14.4.5).

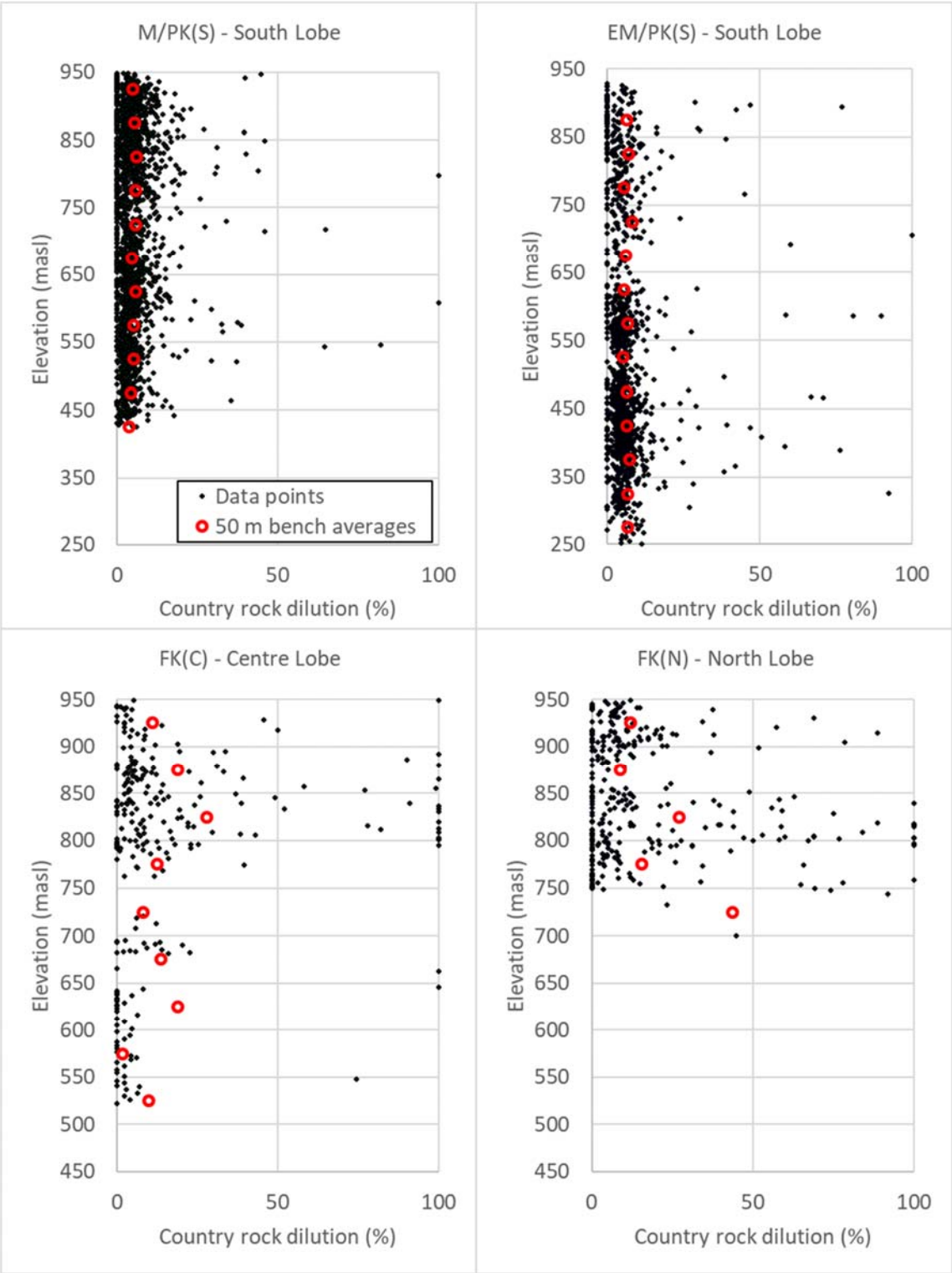

Figure 14-1: Dilution measurements and average dilution per 50 m bench in the volumetrically significant kimberlite units.

Drill core petrography

A large suite of spatially representative petrography samples (n = 227) was collected in 2017. The samples were derived from 13 of the 2017 deep core drill holes (n = 135) and from 10 historical core drill holes (n = 92). A key objective of the petrographic analysis was to assess the degree of continuity with depth in the two major units of the South Lobe (i.e. M/PK(S) and EM/PK(S)). Analysis involved the observation of key textural and component characteristics of the samples, including: structure and packing density, olivine abundance and size range, country rock xenolith abundance, type and size, groundmass mineralogy, and kimberlite indicator mineral abundance and types. This study did not reveal any evidence for large scale variations in any of these parameters within the M/PK(S) or EM/PK(S) units (MSC18/005R).

14.3 Bulk density and tonnage

The bulk density dataset was updated to include new measurements from 2017 drilling and the same interpolation approach used in Oberholzer et al. (2017) was applied to populate local estimates of bulk density into the block model. This is this basis for the bulk density estimates above 604 masl. Below 604 masl the sample coverage does not adequately constrain bulk density on a local basis, but the interpolation does provide a reasonable broad-scale representation of bulk density characteristics with depth. Interpolated bulk density below 604 masl was therefore extracted from GEMS through volumetric reporting to obtain average bulk density by 12 m bench for each domain. Averages by elevation range (selected to encompass any large-scale trends present) were initialized back into the block model.

14.3.1 Data

The bulk density dataset used for this estimate derives from two major phases of work:

- Historical dry bulk density measurements (n = 2,466 internal to the pipe) from early evaluation work between 2003 and 2007 that were used as a basis for all bulk density estimates prior to this report.
- New wet bulk density measurements (n = 347) from drilling carried out in 2017. One measurement was discarded as an outlier (possible data capture error or sample disaggregation in water during measurement) and an additional 4 samples are external to the pipe. Sample results were adjusted for moisture content (Section 12.3) and were integrated with the historical dataset.

The final dataset (n = 2,808) used to estimate bulk density is summarized in Table 14-2 and illustrated spatially in Figure 14-2. The dataset includes data from recent (2017) drilling for which the exact sample location is uncertain due to unreliable or absent down-hole orientation survey results (Section 12.1.1). The use of somewhat uncertain location data for these samples is not considered problematic in the context of the large scale smoothing inherent in the interpolation approach (see Section 14.3.2).

Table 14-2: Dataset used for estimation of bulk density. Bulk density groups illustrate the interpolation approach used – hard boundaries were used between data points in different groups.

Lobe	Domain	Bulk density group	Samples	Bulk density (g/cm^3)			
				Average	Minimum	Maximum	Standard deviation
Centre	BBX(C)	Breccia	67	2.55	2.13	2.88	0.16
	CFK(C)	C/N Primary	156	2.61	2.34	2.81	0.10
	CKIMB(C)	C/N Weathered	8	2.35	1.87	2.60	0.31
	FK(C)	C/N Primary	182	2.57	1.93	2.95	0.20
	KBBX(C)	Breccia	20	2.59	1.96	2.83	0.21
	WK(C)	C/N Weathered	124	2.19	1.80	2.81	0.27
	Centre Total		**557**	**2.49**	**1.80**	**2.95**	**0.25**
North	BBX(N)	Breccia	86	2.53	1.98	2.78	0.17
	CKIMB(N)	C/N Weathered	8	2.26	1.99	2.45	0.18
	FK(N)	C/N Primary	138	2.43	1.87	2.76	0.16
	WBBX(N)	C/N Weathered	9	2.42	2.00	2.71	0.24
	WK(N)	C/N Weathered	50	2.28	1.84	2.63	0.20
	North Total		**291**	**2.43**	**1.84**	**2.78**	**0.19**
South	BBX(S)	Breccia	9	2.73	2.36	2.89	0.18
	CBBX(S)	S Weathered	3	2.19	2.10	2.26	0.08
	CKIMB(S)	S Weathered	19	2.41	1.89	3.04	0.29
	EM/PK(S)	S Primary	311	2.92	2.33	3.25	0.17
	INTBS(S)	Breccia	9	2.36	1.95	2.67	0.24
	M/PK(S)	S Primary	1,261	2.92	1.81	3.23	0.20
	WBBX(S)	S Weathered	74	2.18	1.81	2.88	0.25
	WK(S)	S Weathered	230	2.32	1.80	3.12	0.32
	WM/PK(S)	S Primary	44	2.56	2.27	2.80	0.11
	South Total		**1,960**	**2.80**	**1.80**	**3.25**	**0.32**
AK6 Total			**2,808**	**2.70**	**1.80**	**3.25**	**0.34**

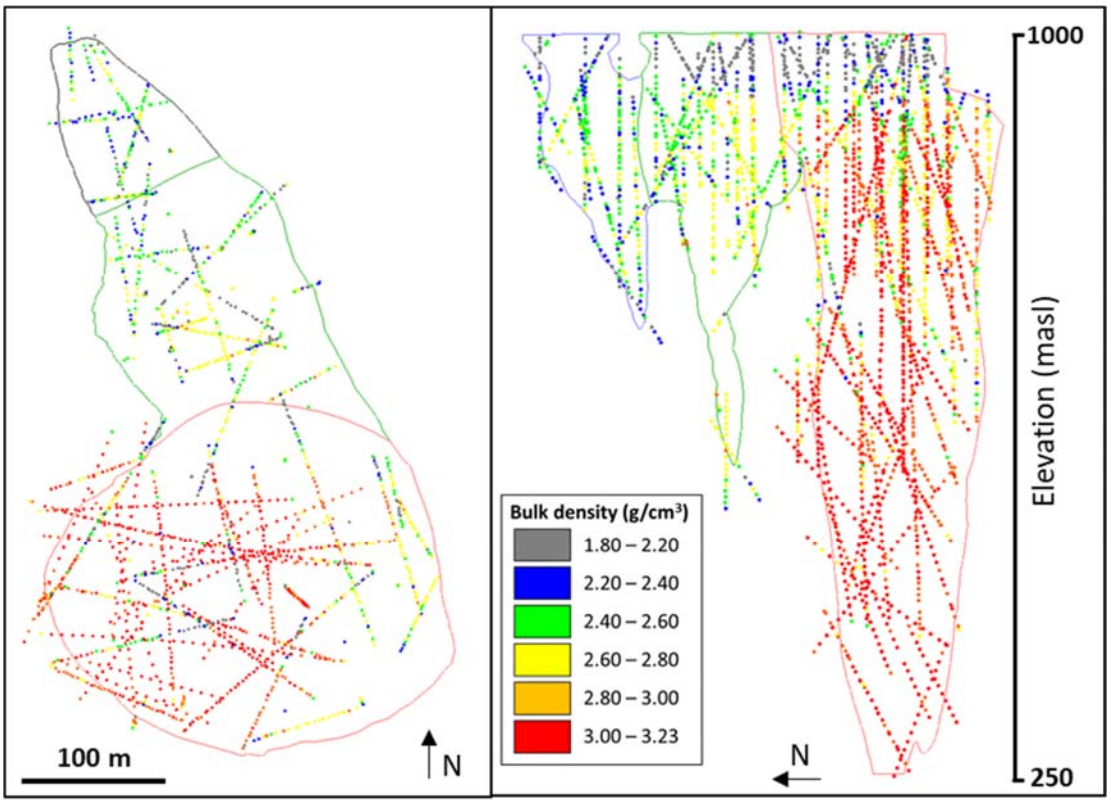

Figure 14-2: Bulk density sample coverage in plan view (left) and vertical section (right). In both cases the lobe outlines are shown on specified planes. In plan view this is the 900 masl elevation and in the vertical section this is a north-south oriented plane. Sample positions have been projected onto these planes and are located within the pipe shell even though they may appear outside in these plots.

14.3.2 Bulk density estimation approach

Bulk density data were combined into sample groups (Table 14-2) based on geology (e.g. lobes; weathered vs. fresh; breccia vs. kimberlite). Model variograms derived by Bush (2008a; Table 14-3), 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 predominantly carried out in a single pass by using the neighbourhood searches shown in Table 14-4. This first pass interpolation resulted in 23,429 bulk density allocations to blocks. A second pass interpolation using a larger search radius of 240, 240, 76 (X, Y, Z) populated a further 86 blocks that were not informed by the first pass.

Table 14-3: Variogram parameters used for bulk density estimation. Expo = exponential, Sph = spherical, range is reported in metres.

BD Group	Nugget	Model	Sill	Range			Model	Sill	Range		
				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	0.010	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

Table 14-4: Neighbourhood parameters used for bulk density estimation.

BD Group	Minimum	Optimal	Search Radii		
			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

Based on the sample coverage available, for the most part this interpolation is considered to provide reliable local bulk density estimates to a depth of 604 masl. The only exception to this is the Centre Lobe FK(C) domain in the elevation range 760 to 676 masl (block levels 23 to 29). Due to poor sample coverage the interpolated bulk density estimates for this zone were primarily informed by samples from FK(N) that have significantly lower bulk density than those of the majority of FK(C) samples. This resulted in interpolated bulk density estimates in the 760 to 676 masl elevation range (2.47 g/cm^3) that are significantly lower than the remainder of the FK(C) domain. Average interpolated block bulk densities of FK(C) in the two benches overlying and the two benches underlying this elevation range are 2.61 and 2.67 g/cm^3 respectively. All FK(C) material in the elevation range 760 to 676 masl was therefore initialized with an average of 2.64 g/cm^3 to correct this localized underestimation of bulk density. No further modifications were made to the interpolated bulk density estimates above 604 masl.

Below 604 masl the sample coverage, while spatially representative, is more dispersed and does not adequately constrain bulk density on a local basis. Interpolated results below 604 masl were extracted by volumetric reporting from the block model for each domain by 12 m bench, as shown in Figure 14-3. A slight increase in bulk density with depth is present in the EM/PK(S) domain from 604 to 520 masl. There is no significant trend in the M/PK(S) domain with depth below 604 masl. Average bulk densities by domain were initialized into the block model by elevation range as shown in Table 14-5.

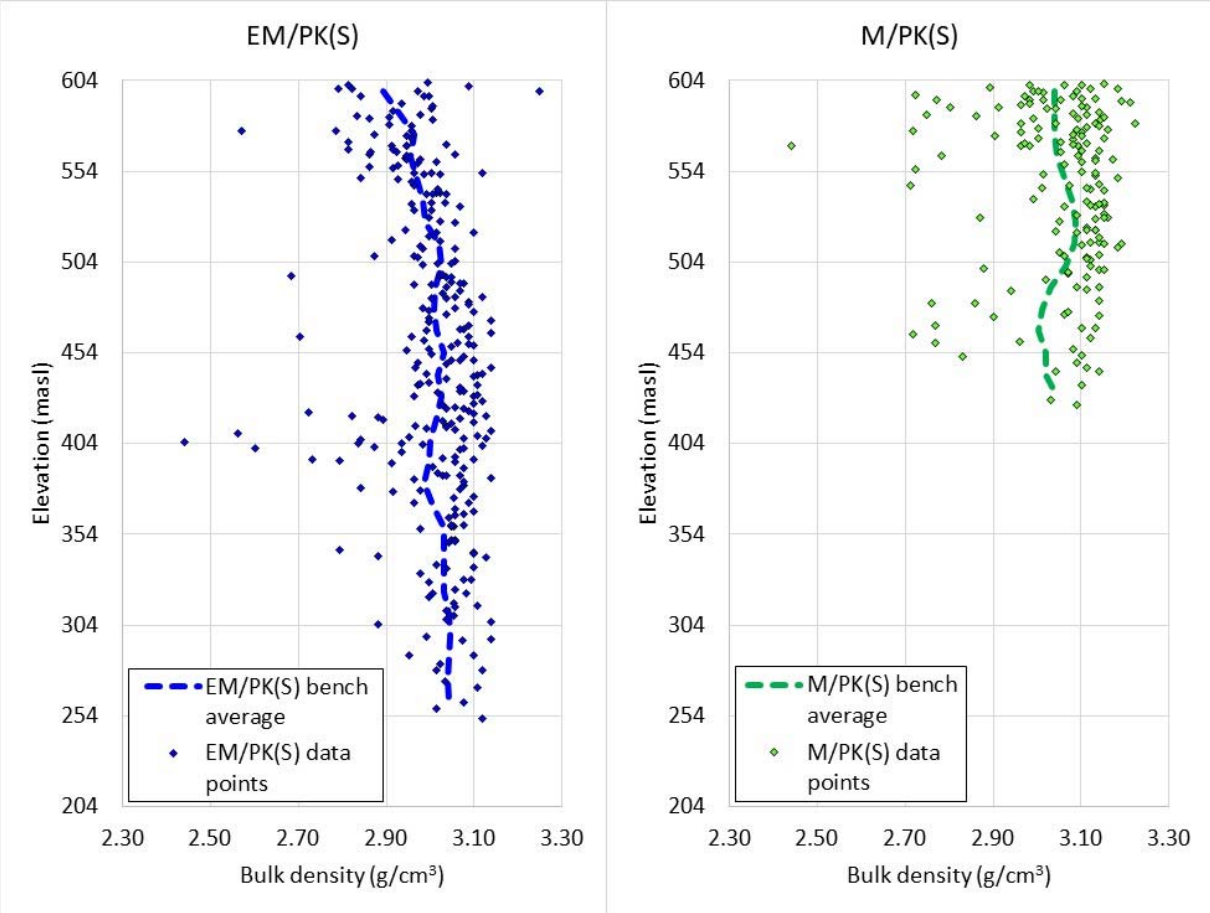

Figure 14-3: Average interpolated block bulk density per domain by 12 m bench below 604 masl. The bulk density sample data on which the interpolations are based are shown for reference.

Table 14-5: Bulk density estimates below 604 masl.

Domain	Elevation (masl)		Sample data		Interpolated average BD extracted from block model (g/cm^3)
	From	To	Samples	Average BD (g/cm^3)	
EM/PK(S)	604	592	11	2.95	2.89
	592	580	9	2.93	2.93
	580	568	12	2.89	2.96
	568	556	15	2.93	2.95
	556	544	10	2.98	2.97
	544	532	11	3.01	2.98
	532	520	6	3.02	2.99
	520	256	197	3.02	3.02
M/PK(S)	604	256	158	3.05	3.05

14.3.3 Summary of bulk density and tonnage estimates

Estimates of volumes, tonnes and average bulk densities by bulk density group (as extracted from the block model by volumetric reporting) are provided in Table 14-6. The variation in average bulk density with depth by lobe is illustrated in Figure 14-4. Bulk density block models for the EM/PK(S) and M/PK(S) domains are illustrated in Figure 14-5.

Table 14-6: Summary of bulk density and tonnage estimates. Results are shown by bulk density group. Note that the M/PK(S) and EM/PK(S) belong to the same bulk density group (South primary, as per Table 14-1).

Lobe	Bulk density group	Volume (Mm^3)	Bulk density (g/cm^3)	Tonnage (Mt)
North	Breccia	0.29	2.54	0.73
	Kimberlite	0.43	2.45	1.04
	Weathered	0.40	2.28	0.92
	North Total	**1.12**	**2.41**	**2.69**
Centre	Breccia	0.35	2.56	0.89
	Kimberlite	2.28	2.58	5.87
	Weathered	0.92	2.21	2.02
	Centre Total	**3.54**	**2.48**	**8.79**
South	Weathered	2.45	2.28	5.59
	Breccia	0.17	2.58	0.45
	EM/PK(S)	7.47	2.92	21.81
	M/PK(S)	12.14	2.93	35.59
	South Total	**22.23**	**2.85**	**63.45**
Total AK6		**26.89**	**2.79**	**74.93**

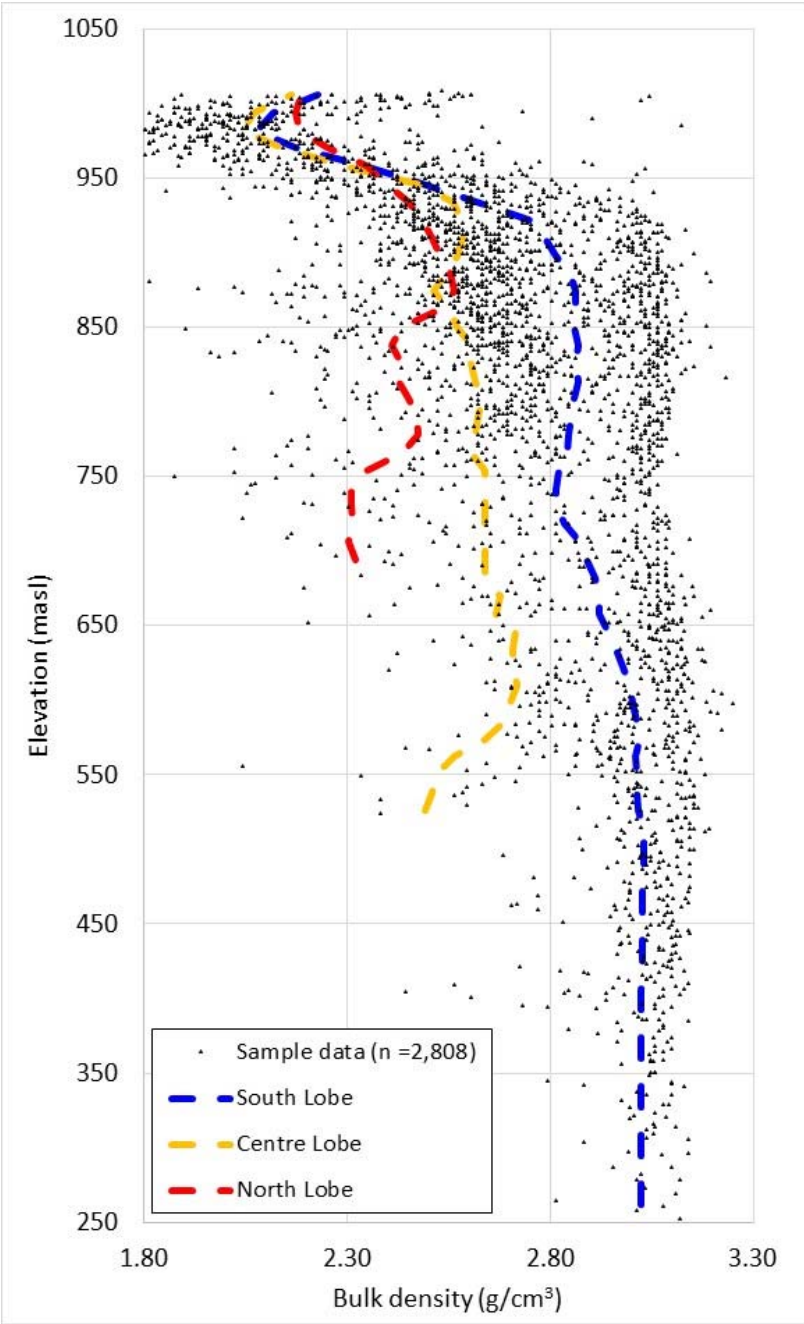

Figure 14-4: Interpolated bulk density estimates (12 m bench averages) by lobe. Supporting bulk density sample data are shown for reference.

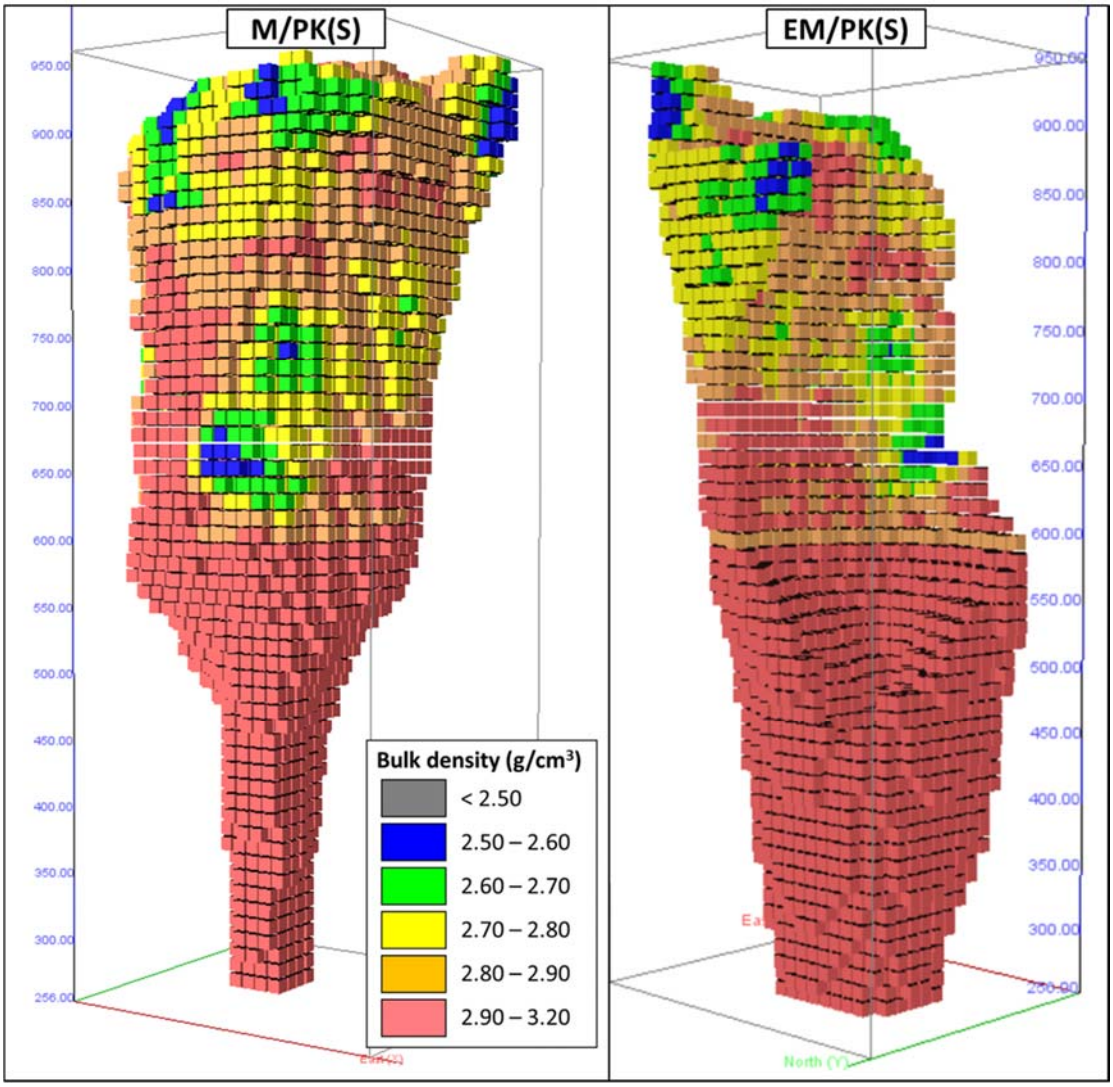

Figure 14-5: Bulk density estimates as hosted in the block model for the M/PK(S) and the EM/PK(S) domains. Interpolated local estimates were used above 604 masl. Averages by elevation range derived from interpolations were used below 604 masl.

14.4 Grade

14.4.1 Supporting data – macrodiamonds

Two large diameter drill (LDD) sampling programs were carried out in two phases from 2006 to 2007, during which a total of 25 holes comprising 7,964 m of 23 inch diameter drilling were completed. Samples comprising 12 m increments down hole were collected and processed from 24 of these LDD holes. The holes were drilled vertically and are well-distributed across the pipe (Figure 14-6). LDD diamond recoveries are summarised by lobe and DTC size class in Table 14-7.

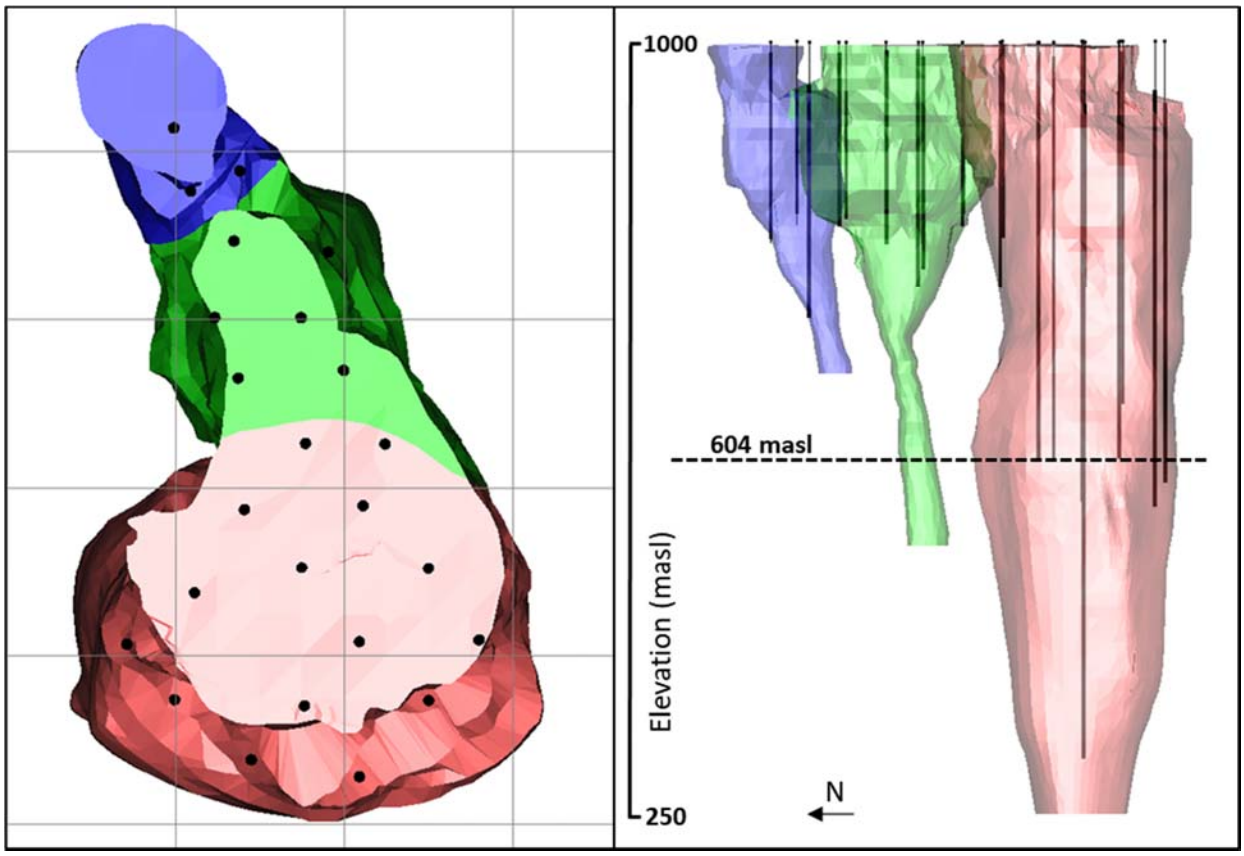

Figure 14-6: LDD drill hole coverage for the 24 LDD holes used for grade estimation. The thick portion of the drill trace reflects sample coverage.

Table 14-7: LDD recoveries by lobe. This dataset includes all sample results from within the revised shell model and excludes diamond recoveries from spillage and any results not attributable to specific samples (e.g. plant clean up). Additional results (from LDD027) sourced in 2018 are included. This table therefore differs slightly from that presented in Oberholzer et al. (2017, Table 14.8). These results do not reflect the results of the grade capping exercise referred to later in this section.

Size	North Lobe		Centre Lobe		South Lobe	
	St	Ct	St	Ct	St	Ct
DTC1	222	3.81	928	16.96	3,954	69.83
DTC3	249	8.72	748	26.32	3,528	121.93
DTC5	260	21.35	484	38.87	2,443	187.98
DTC7	98	12.20	194	23.98	780	94.53
DTC9	75	15.54	167	34.36	523	106.98
DTC11	58	26.84	85	34.13	287	115.47
DTC13	15	12.21	27	22.60	76	62.29
DTC15	3	2.35	6	6.06	13	15.04
DTC17	7	9.13	7	10.61	16	23.25
DTC19	1	2.27	6	13.47	25	54.68
DTC21	0	0.00	0	0.00	9	35.18
DTC23	0	0.00	1	13.37	2	7.98
Totals	**988**	**114.42**	**2,653**	**240.73**	**11,656**	**895.13**
Samples	**45**		**108**		**420**	
Volume (m^3)	**152**		**368**		**1,404**	
cpm^3	**0.75**		**0.65**		**0.64**	

Sample volumes were measured by caliper surveys of all holes. Volumes per sample vary from a minimum of 0.27 to a maximum of 5.31 m^3, with an average of 3.36 m^3. Samples were assigned bulk density values derived from the average bulk density of the block in which the mid-point of each sample is located. Sample bulk densities are therefore consistent with those reported in Section 14.3.

The De Beers grade dataset from 2008 (Bush, 2008a) was coded to the updated geology model solids. LDD samples have been included or excluded where they fall inside or outside of the remodelled pipe shell, yielding the same grade dataset used in Lynn et al. (2013). Comparison of available LDD records (carats and volume) with the 2008 dataset indicates that the latter closely approximates but does not exactly match the +2DTC sample grades (average deviation of ~1 %). In the absence of detailed documentation on historical work it is not possible to verify the reason for this minor discrepancy but for the sake of consistency with historical estimates and in view of the negligible impact on estimated grades, the original grade dataset has been used for the estimate reported here. Additional sample results from LDD027 in the depth range 396 to 702 m downhole (617 to 311 masl) have recently been identified and 5 additional sample results (617 to 557 masl) from LDD027 were added to the grade dataset.

Review of interpolated block model grades derived using this dataset highlighted grade anomalies where outlier sample grades result in unrealistically high local grades within the block model. A capping exercise was therefore carried out. A graphical approach was used whereby grades beyond visually identified

inflection points (selected based on ranked plots of sample grade) were capped to the highest grade value below the selected cut-off. For the North and South Lobes this was undertaken based on the grade groups used for interpolation (see Table 14-8). Because of the significant difference in grade between M/PK(S) and EM/PK(S), grade data from the South Lobe domains were grouped on a more detailed basis for the capping exercise (Table 14-8). This grade capping approach is similar to but slightly more aggressive than that reported in Oberholzer et al. (2017) and has resulted in the capping of 28 of 535 grade values (Table 14-9) used for interpolation. The final grade estimation dataset is summarised by grade group in Table 14-8.

Table 14-8: Summary of the grade dataset used to interpolate local estimates of grade above 604 masl.

Grade group	Sub-group	Samples	Average cpm^3	Minimum cpm^3	Maximum cpm^3
South breccia	N/a	13	0.20	0.00	0.61
South primary	M/PK(S), WM/PK(S)	226	0.44	0.02	1.26
	WK(S), CKIMB(S)	63	0.43	0.00	1.41
	EM/PK(S)	86	0.81	0.06	1.41
Centre/North primary	N/a	125	0.63	0.06	1.86
Centre/North breccia	N/a	22	0.46	0.00	1.56

Table 14-9: Sample grade capping carried out prior to interpolation of grade. Capping thresholds were selected based on visually selected inflection points in ranked plots of sample grade.

Sample ID	Lobe	Domain	Sample cpm^3	Capped cpm^3
DCD673	North	FK(N)	2.06	1.86
DCD662to665		WK(N)	4.18	1.86
DCE474		WKBBX4(N)	2.31	1.56
DCD652		WKBBX4(N)	2.25	1.56
DCE146	Centre	CFK(C)1	3	1.86
DCD710		CFK(C)1	2.82	1.86
DCD603		CFK(C)1	2.06	1.86
DCE183		FK(C)	2.15	1.86
DCD758	South	EM/PK(S)	7.36	1.41
DCE227		EM/PK(S)	4.24	1.41
DCD760		EM/PK(S)	2.7	1.41
DCE230		EM/PK(S)	2.21	1.41
DCE234		EM/PK(S)	1.9	1.41
DCD756		EM/PK(S)	2.44	1.41
DCD804		EM/PK(S)	1.66	1.41
DCE243		EM/PK(S)	1.54	1.41
DCD786to787		M/PK(S)	2.76	1.26
DCD781		M/PK(S)	9.03	1.26
DCE286		M/PK(S)	1.95	1.26
DCD753		M/PK(S)	1.52	1.26
DCD750		M/PK(S)	1.5	1.26
DCE396		M/PK(S)	1.47	1.26
DCD626		M/PK(S)	1.43	1.26
DCE257		WBBX(S)	0.72	0.61
DCD722		WK(S)	4.93	1.41
DCD778		WK(S)	2.25	1.41
DCD741		WK(S)	1.91	1.41
DCD619		WKBBX(S)	2.16	1.26

The newly remodelled EM/PK(S) domain (Section 7.4.2) is exposed at surface in the open pit, and a controlled production run sourced from within this domain was carried out between 9 and 20 February 2018. During this period a total of 79,052 tonnes were processed, from which 13,562 ct were recovered (Table 14-10) yielding a grade of 0.17 cpt. Quality control procedures in place for this exercise are discussed in Section 12.6.2; the sample is considered to exclusively represent EM/PK(S) with no scope for significant contamination.

Table 14-10: Diamond recoveries from the EM/PK(S) controlled production run carried out between 9 and 20 February 2018. The number of diamonds in the DTC11 and smaller size classes are not recorded during production.

Screen size (mm)	Stones	Carats	Percent carats
+10.8 ct	44	1,265	9.3
DTC 23	38	302	2.2
DTC 21	180	859	6.3
DTC 19	302	771	5.7
DTC 17	223	331	2.4
DTC 15	174	187	1.4
DTC 13	1166	901	6.6
DTC 11	n/a	2,366	17.4
DTC 9	n/a	1,915	14.1
DTC 7	n/a	1,489	11.0
DTC 5	n/a	2,612	19.3
DTC 3	n/a	547	4.0
DTC 1	n/a	18	0.1
Total		**13,562**	

The Karowe Mine has been in production since 2012 and comprehensive records of grade, SFD and diamond value are available for all production (summarised in Table 6-1). Karowe maintains detailed haulage records documenting the source, stockpile location and production date for all kimberlite material (Section 12.6.1). Recent production records have been used to define the recovery efficiency of the Karowe process plant (Section 14.4.7).

14.4.2 Supporting data – microdiamonds

A comprehensive microdiamond sampling program was carried out on cores from 2017 drilling that targeted the South Lobe in the elevation range below 600 masl (Section 10.2). The samples each comprise approximately 8 kg of drill core and were collected at a spacing of ~5 m down hole to achieve a broad, spatially representative coverage of the pipe in the elevation range 600 to 256 masl. Additional sampling from historical pilot holes in the South Lobe was carried out to obtain representative microdiamond results for kimberlite sampled by LDD drilling. This was required to constrain the relationship between microdiamond stone frequency (stones per kilogram) and grade (carats per tonne) (see Section 14.4.6). A total of 916 aliquots weighing 7,315 kg were collected and processed at the Saskatchewan Research Council (SRC) Geoanalytical Laboratories in Saskatoon, Canada (Sections 11 and 12). Historical microdiamonds results (77 aliquots weighing 1,436 kg) collected and processed prior to 2010 were not used in this estimate. These samples were collected from core and RC chip material, and the process and QA/QC methodology are not known. Comparison of historical results with new results shows significant inconsistencies between the datasets. As a result, the historical data were not used in the estimation process. Total microdiamond recoveries by sieve class for the 2017/2018 dataset used in this estimate are provided in Table 14-11 and the spatial sample coverage is illustrated in Figure 14-7.

Table 14-11: Total microdiamond recoveries by standard CIM sieve class for the 2017/2018 dataset used to support grade estimates at AK6. Total sample mass was 7,315 kg.

Screen size (mm)	Stones	Carats
0.105	1,440	0.033
0.150	926	0.059
0.212	614	0.106
0.300	426	0.220
0.425	233	0.359
0.600	141	0.614
0.850	63	0.726
1.180	40	1.341
1.700	8	0.766
2.360	1	0.394
Total	**3,892**	**4.618**

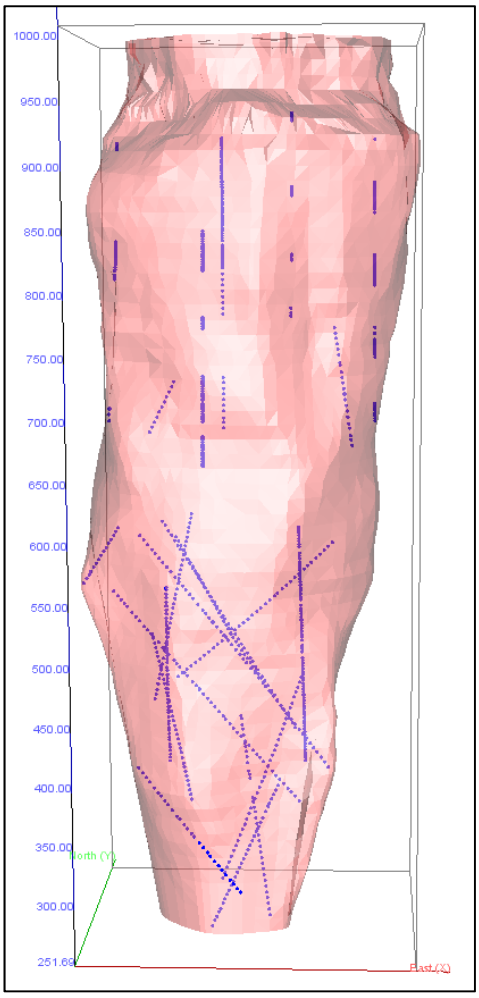

Figure 14-7: Microdiamond sample coverage. Each blue dot represents an ~8 kg sample aliquot.

14.4.3 Macrodiamond stone frequency and SFD characteristics

A thorough investigation of macrodiamond stone frequency and SFD characteristics (MS13/023R) was carried out in support of the resource update reported in Lynn et al. (2014). It was found that the SFD of the LDD parcels in each lobe reflected the differences in production SFD between lobes, and the LDD data were assessed for any indication of a change in SFD with depth. No significant changes were noted, and, in conjunction with demonstrated geological and microdiamond SFD continuity, this was used as a basis for the assumption of constant diamond value with depth in each lobe. The updated Mineral Resource Estimate for the deep portion of the South Lobe (below 604 masl) reported here is premised on continuity in grade and SFD within the main domains present (M/PK(S) and EM/PK(S). This section therefore focusses on macrodiamond stone frequency and SFD characteristics in these two domains.

Macrodiamond sample stone frequency data (number of plus DTC3 stones per tonne) grouped by domain and by elevation ranges with depth in the pipe are shown in Figure 14-8. Outlier values (more than 3 standard deviations from the mean) are excluded. In samples of limited size the number of diamonds per unit mass is considered a more reliable indication of grade than the weight of diamonds per unit mass, which is typically more variable and over-influenced by sporadic recoveries of larger diamonds. The results indicate broad large-scale consistency in macrodiamond stone frequency with depth within the M/PK(S) domains. EM/PK(S) yields consistently higher average grades than M/PK(S) and displays an apparently higher degree of variability (approximately +/- 15 %). The data do not show any indication of any large-scale trends, however, and the observed variations are considered to partly reflect the relatively small number of samples available for several of the elevation ranges.

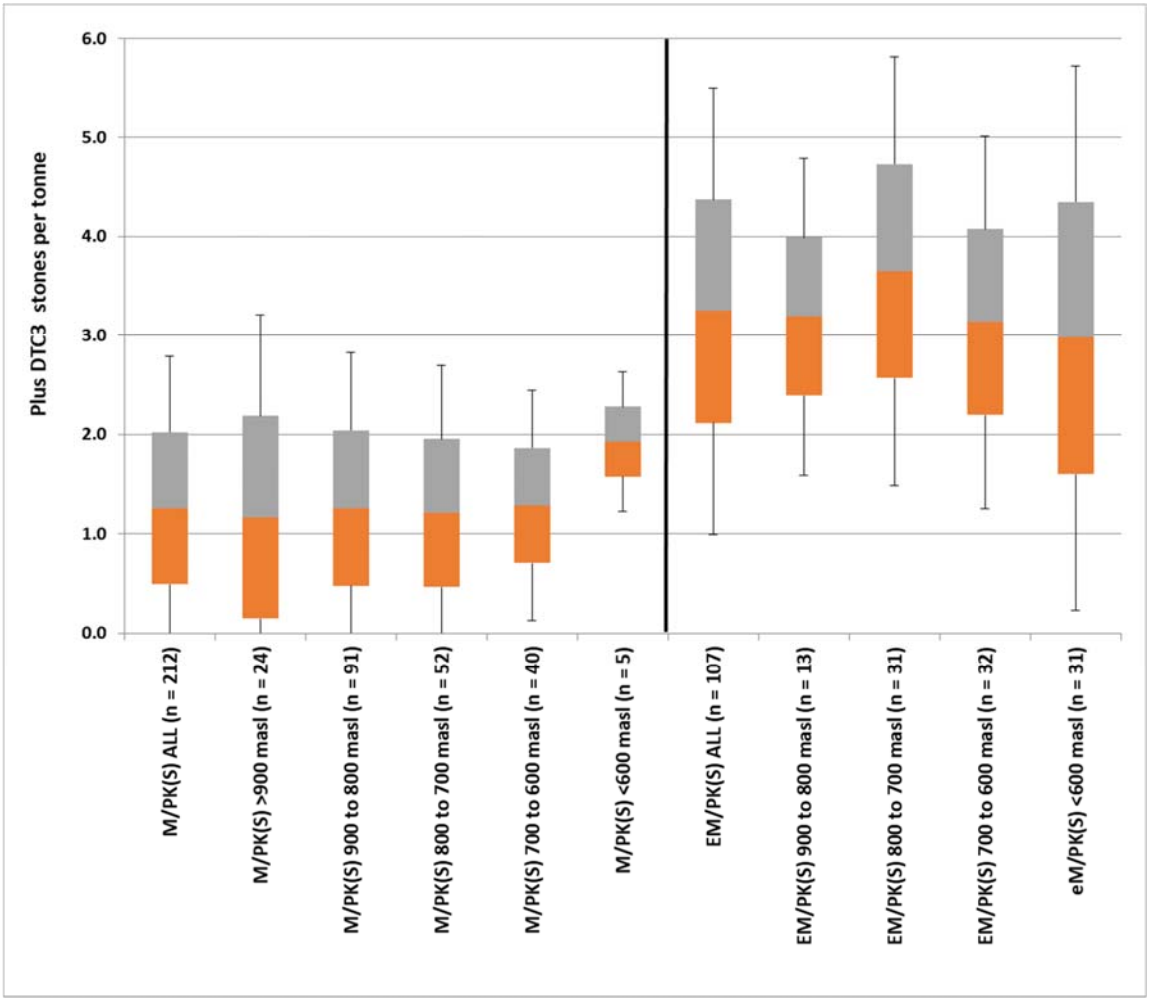

Figure 14-8: Box and whisker plots illustrating variation in plus DTC 3 macrodiamond stone frequencies from LDD samples grouped by domain into broad elevation zones. The combined grey and orange boxes indicate the +1 and -1 standard deviation ranges, respectively, and the contact between them is the mean. Error bars represent the +2 and -2 standard deviation ranges. The number of samples represented by each grouping is indicated in parentheses.

The SFDs of the parcels as grouped in Figure 14-8 are shown in Figure 14-9. Subtle differences in SFD are evident but can largely be attributed to minor variations in diamond recovery efficiency. For example: (1) M/PK(S) above 900 masl reflects a higher content of fine diamonds, attributable to more efficient liberation from weathered material close to surface; and (2) EM/PK(S) below 600 masl appears slightly coarser grained, likely reflecting increasing competency with depth and corresponding less efficient liberation and recovery of finer diamonds. The coarse ends of the defined distributions are variable – reflecting erratic recovery of large diamonds in these relatively small parcels.

Overall, the LDD results indicate broad-scale consistency in macrodiamond stone frequency and SFD with depth in both domains. This corresponds well with observations of large scale geological continuity on the basis of drill core logging and petrographic work (Section 7.3.4).

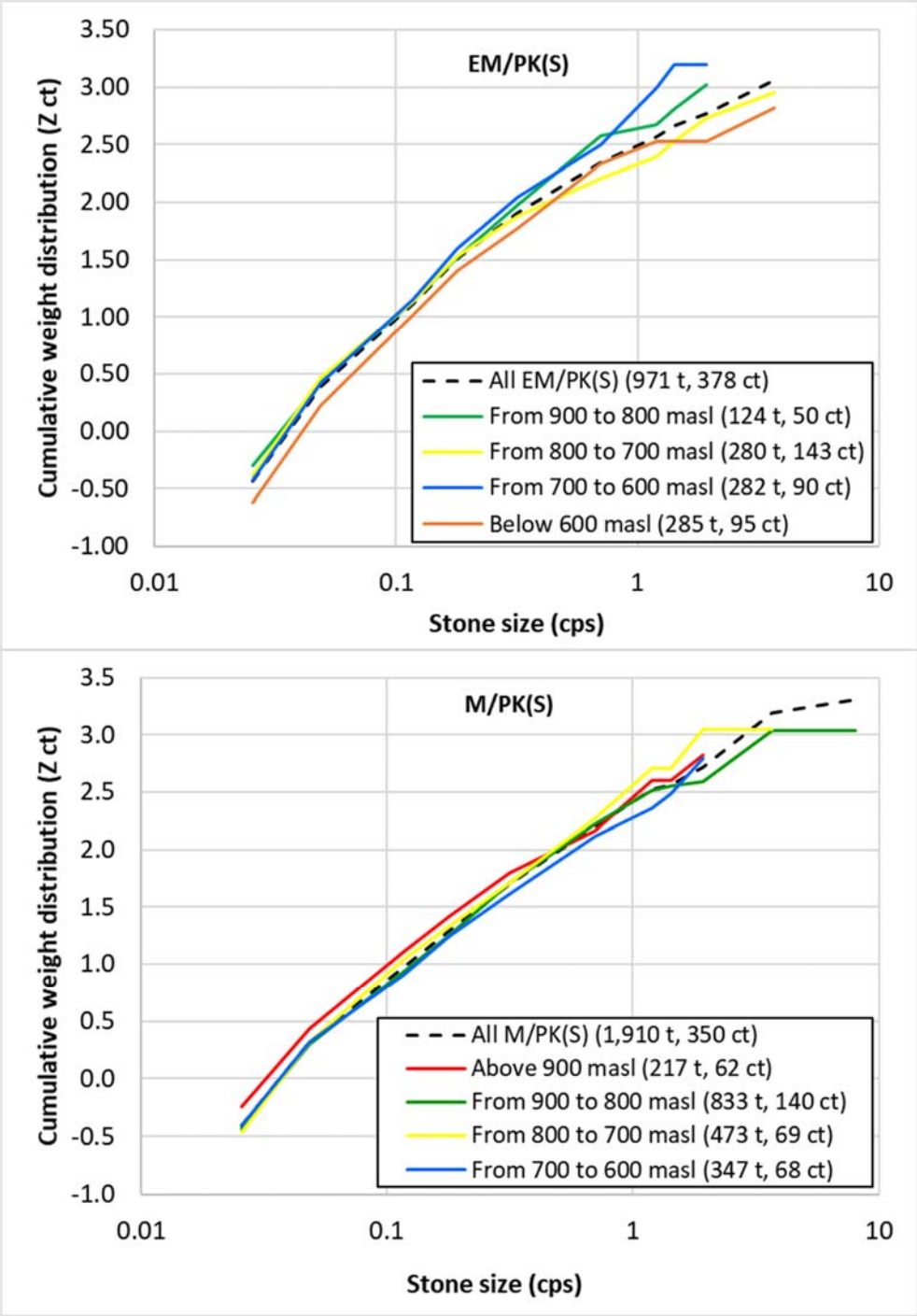

Figure 14-9: Macrodiamond SFDs (+1 DTC) for the EM/PK(S) and M/PK(S) domains grouped into elevation ranges. SFD is shown on a cumulative log probability plot; representing the proportion of carats, expressed as Z values (number of standard deviations from mean assuming a normal distribution), below a given stone size. cps = carats per stone.

14.4.4 Microdiamond stone frequency and SFD characteristics

As for Section 14.4.3, this section focusses on the M/PK(S) and EM/PK(S) domains in support of the estimation approach applied to update the Mineral Resource Estimate for the deep portion of the South Lobe.

Microdiamond stone frequency sample data (number of plus 0.15 mm stones per kilogram) grouped by domain and by elevation ranges with depth are shown in Figure 14-10. Outlier values (more than 3 standard deviations from the mean) are excluded. Despite the relatively small parcels represented, the results show that the M/PK(S) and EM/PK(S) domains display large-scale consistency in stone frequency with depth. EM/PK(S) presents a consistently higher stone frequency than M/PK(S), which is also reflected in the macrodiamond results presented in Section 14.4.3.

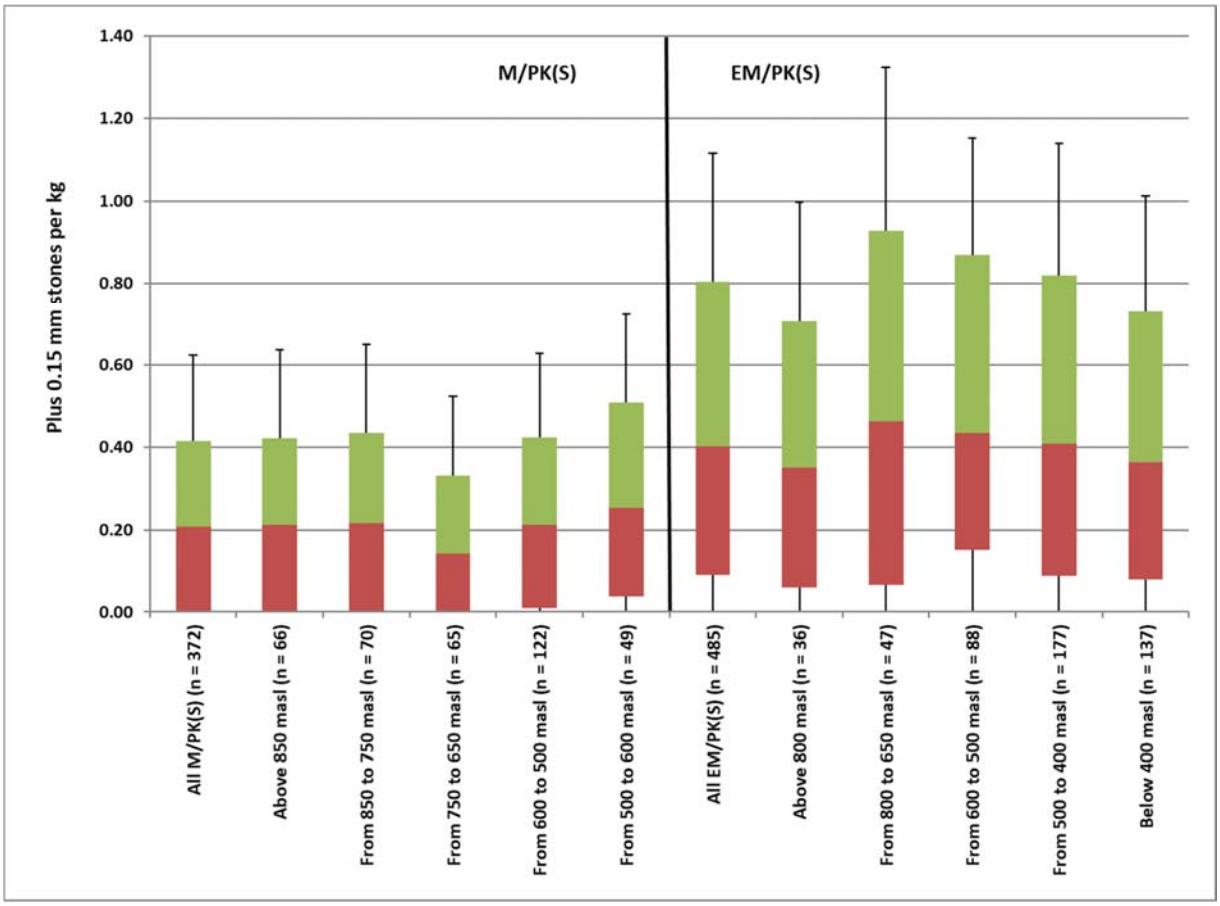

Figure 14-10: Box and whisker plots illustrating variation in plus 0.15 mm microdiamond stone frequencies from drill core samples grouped by domain into broad elevation zones. The combined green and red boxes indicate the +1 and -1 standard deviation ranges, respectively, and the contact between them is the mean. Error bars represent the +2 and -2 standard deviation ranges. The number of microdiamond samples represented by each grouping is indicated in parentheses. The M/PK(S) parcel from 750 to 650 masl derives from a limited number of holes predominantly close to the margin of the pipe and is not representative of the M/PK(S) domain as a whole in that elevation range.

The SFDs of the parcels as grouped in Figure 14-10 are shown in Figure 14-11. Despite the relatively small parcel sizes the results indicate broad consistency in microdiamond SFD with depth in these domains. The 650 to 750 masl grouping for M/PK(S) displays a potentially finer grained SFD than those defined by the other elevation ranges. This grouping of results derives from a limited number of holes predominantly in close proximity to the shell margin, and it is not representative of the M/PK(S) domain as a whole in that elevation range.

Microdiamond results indicate broad-scale consistency in stone frequency and SFD with depth in the M/PK(S) and EM/PK(S) domains. This corresponds well with observations of large scale geological continuity (Section 14.2.2) and with observations of macrodiamond stone frequency and SFD continuity discussed in Section 14.4.3.

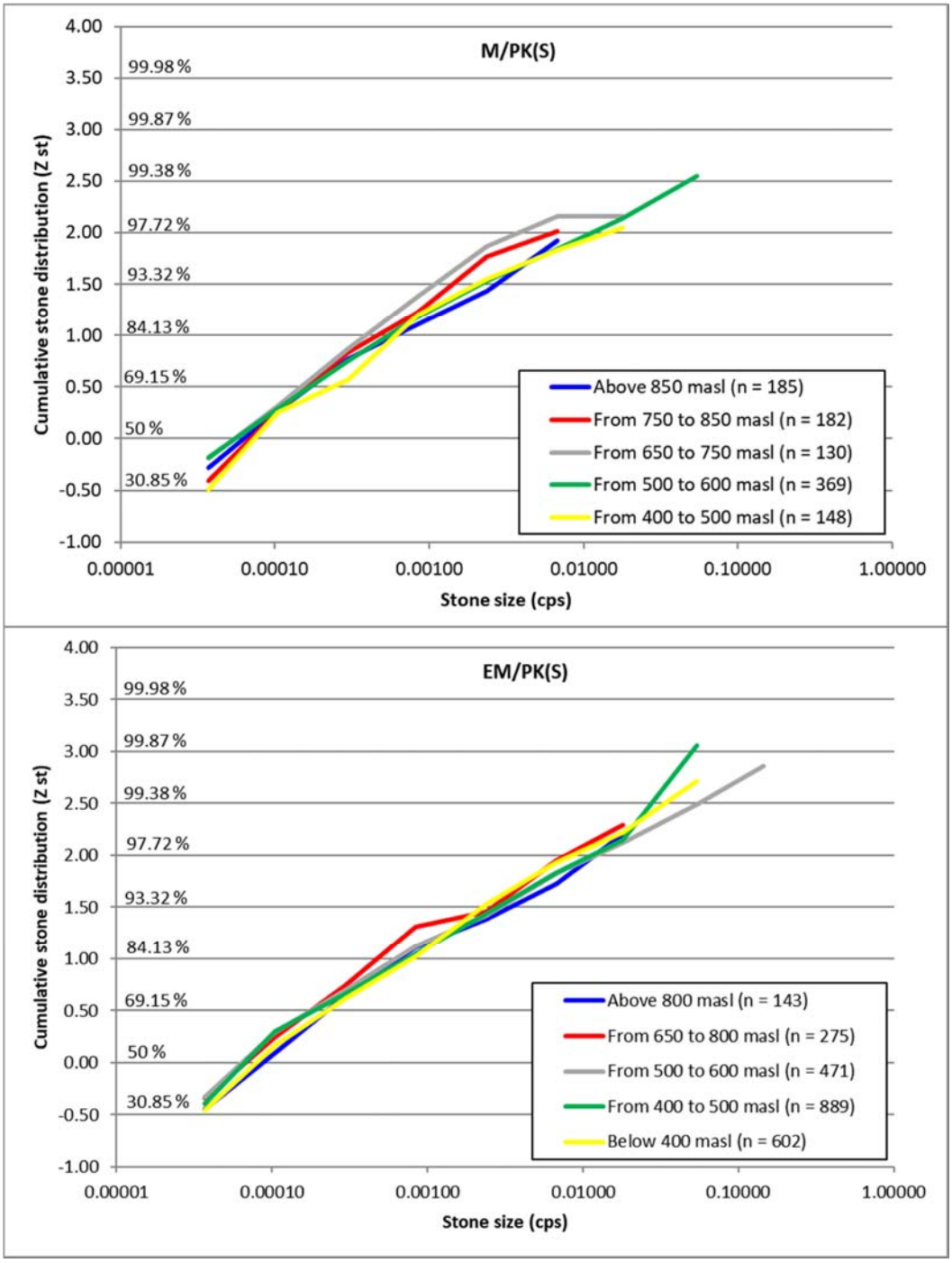

Figure 14-11: Microdiamond SFDs (+0.105 mm) for the EM/PK(S) (above) and M/PK(S) (below) domains grouped into elevation ranges. SFD is shown on a cumulative log probability plot; representing the proportion of carats, expressed as Z values (number of standard deviations from mean assuming a normal distribution), below a given stone size. cps = carats per stone.

14.4.5 Grade estimate above 604 masl

A local grade estimation approach (duplicated from Oberholzer et al. 2017) has been applied from surface to 604 masl where a spatially representative coverage of LDD sampling allows for interpolation of the LDD (+1.0 mm) sample grades into the block model. The grade data (in carats per cubic metre; cpm^3) were combined into groups (Table 14-8) on the basis of geology and grade sample statistics. In contrast to the bulk density analysis, grade groups did not distinguish equivalent weathered and fresh kimberlite units (i.e. these were included in the same groups).

The variogram and neighbourhood parameters determined by Bush (2008a; Tables 14-12 and 14-13) were used as inputs for local grade estimation by ordinary kriging. The grade dataset used was modified slightly from that used in previous estimates through the addition of 5 extra grade points from LDD027 (Section 14.4.1, these data were recently sourced in 2018) and through grade capping of outlier grade values that were found to over-influence the interpolated grade estimates on a local basis (Section 14.4.1 Table 14-9). There are insufficient data from the breccia units for variography (Bush, 2008a). Thus, the variograms for the equivalent primary kimberlite grade groups were used for the breccia units in 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” (grade values were interpolated across these boundaries). Two kriging passes with different search neighbourhoods were carried out for each group (Table 14-13). The second pass comprised a larger search neighbourhood and was used to populate blocks uninformed from the first pass. A summary of the number of blocks interpolated through each stage of this process is provided in Table 14-14.

Table 14-12: Variogram parameters for grade estimates above 604 masl.

Lobe	Nugget	Model	Sill	Range		
				X	Y	Z
South	0.120	Spherical	0.175	115	115	83
Centre/North	0.172	Spherical	0.133	90	90	77

Table 14-13: Neighbourhood parameters for grade estimates above 604 masl.

Lobe	Interpolation pass	Minimum samples	Optimal samples	Search Radii		
				X	Y	Z
South	First	3	10	100	100	48
	Second	3	10	150	150	96
Centre/North	First	3	10	100	100	60
	Second	3	10	150	150	108

Table 14-14: Summary of the number of blocks informed with grade estimates through each interpolation run.

Grade group	Domains included	Interpolation pass	Blocks informed
South primary	M/PK(S), EM/PK(S), CKIMB(S), WK(S), WM/PK(S)	1	10,827
		2	3
South breccia	WBBX(S), WKBBX(S), BBX(S), CBBX(S), KBBX(S), CKBBX(S)	1	801
		2	167
Centre/North primary	CKIMB(N), WK(N), FK(N), CKIMB(C), WK(C), FK(N)	1	3,041
		2	816
		N/a[1]	212
Centre/North breccia	WBBX2(N), WBBX5(N), WBBX9(N), WKBBX4(N), BBX1(N), BBX2(N), BBX4(N), BBX5(N), BBX7(N), BBX8(N), BBX9(N), KBBX1(N), KBBX4(N), WBBX5(C), BBX1(C), BBX2(C), BBX3(C), BBX4(C), BBX5(C), BBX6(C), BBX9(C), KBBX1(C), KBBX2(C), KBBX3(C)	1	838
		2	547
		N/a[2]	1

[1] Deep portion of FK(C) not fully informed below 700 masl, average interpolated block grade from 736 to 700 masl (0.512 cpm^3) was applied to all FK(C) material below 700 masl.

[2] Single block not informed by second pass - assigned average block grade from its 12 m bench (1.135 cpm^3).

Grade estimates above 604 masl are summarised by lobe and grade group in Table 14-15 and are illustrated graphically in Figure 14-12. Major domains of the South Lobe primary grade group are presented separately in Table 14-15 to illustrate the significant grade difference between M/PK(S) and EM/PK(S). The Centre Lobe primary grade group in Table 14-15 includes a very limited volume (0.08 million m^3) of FK(C) below 604 masl. Due to the limited volume below 604 masl the grade of this material was not estimated separately. Note that these grades are estimated as +1 mm LDD-recoverable carats per cubic metre[1] and require adjustment for current process plant recovery efficiency at a bottom cut-off 1.25 mm (Section 14.4.7). Recoverable volume-based grades (cpm^3) are integrated with the bulk density estimates in the block model to generate estimates of recoverable carats per tonne.

[1] Samples were processed at a 1 mm bottom cut off and these estimates are therefore referred to as those recoverable at +1 mm. The grade interpolation dataset used is however most closely approximated by +2DTC sample grades with the +1DTC size fraction excluded.

Table 14-15: Summary of grade estimates (+1 mm LDD recoverable cpm^3) above 604 masl. The major domains EM/PK(S) and M/PK(S) are separated from the weathered and breccia groupings in the South Lobe to illustrate the significant difference in grade between them. Mm^3 = million cubic metres, Mct = Million carats, cpm^3 = carats per cubic metre.

Lobe	Grade group	Domains	Volume (Mm^3)	Carats (Mct)	+1 mm grade (cpm^3)
North	Centre/North breccia	All breccia	0.34	0.16	0.47
	Centre/North primary	All primary	0.78	0.55	0.70
Centre	Centre/North breccia	All breccia	0.36	0.15	0.41
	Centre/North primary[1]	All primary	3.18	1.99	0.62
South	South breccia	All breccia	0.63	0.12	0.19
	South primary	EM/PK(S)	2.56	1.87	0.73
		S_MPK	9.51	4.37	0.46
		S_WX	2.00	0.76	0.38
	Total AK6 above 604 masl		**19.36**	**9.96**	**0.51**

[1] The Centre Primary grade group includes a small volume of FK(C) material below 604 masl.

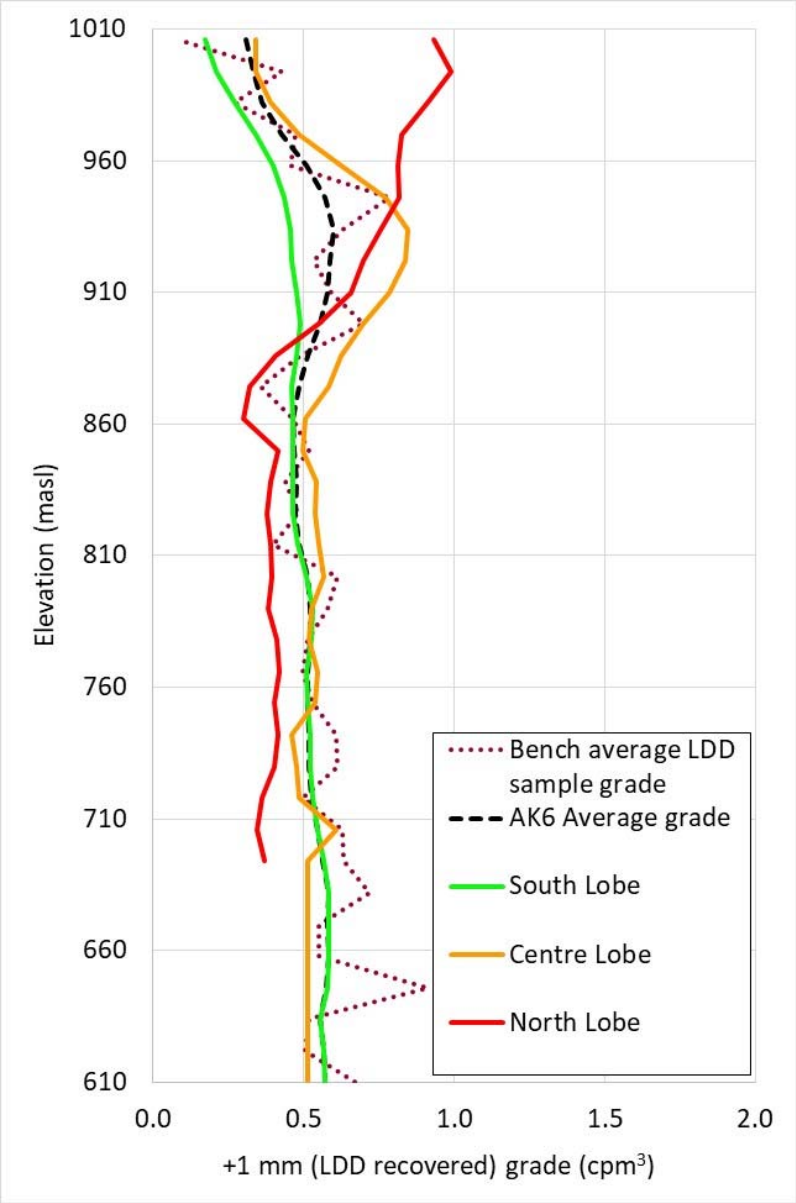

Figure 14-12: Summary of +1 mm cpm^3 grade estimates above 604 masl, as extracted from the AK6 block model by 12 m bench.

14.4.6 Grade estimate below 604 masl

The grade estimate for the South Lobe below 604 masl is based on two components: (1) total content diamond SFD models that define the relationship between microdiamond stone frequency and macrodiamond grade for each of the two resource domains; and (2) a spatially representative dataset of microdiamond samples representing the South Lobe below 604 masl, supporting estimation of microdiamond stone frequency and, in conjunction with the established total content SFD curves, estimation of macrodiamond grades.

Total content SFD models

All available pilot hole drill core adjacent to LDD holes was sampled to facilitate calibration of the relationship been microdiamond stone frequency in drill core and +1 mm macrodiamond grade from LDD samples, as represented by total diamond content SFD curves. For this purpose, adjacent microdiamond and macrodiamond sample results were accumulated into parcels by domain (Table 14-16). The locations from which these parcel groupings derive are shown in Figure 14-13.

Table 14-16: Microdiamond and LDD macrodiamond datasets used to generate total diamond content SFD curves for the M/PK(S) and EM/PK(S) domains.

LDD macrodiamonds					Drill core microdiamonds				
Domain	EM/PK(S)		M/PK(S)			EM/PK(S)		M/PK(S)	
Mass (t)	233		274			0.914		1.548	
Screen size	St	Ct	St	Ct	Screen size (mm)	St	Ct	St	Ct
DTC 1	290	5.01	124	2.19	0.105	178	0.0040	184	0.0040
DTC 3	296	10.07	137	4.90	0.150	126	0.0080	106	0.0063
DTC 5	242	16.77	108	7.98	0.212	97	0.0161	83	0.0138
DTC 7	72	8.78	31	4.04	0.300	74	0.0358	50	0.0249
DTC 9	49	9.84	19	4.17	0.425	21	0.0321	27	0.0377
DTC 11	28	12.20	13	4.71	0.600	15	0.0711	13	0.0589
DTC 13	2	1.13	2	1.92	0.850	17	0.1868	8	0.1101
DTC 15	0	0.00	0	0.00	1.180	4	0.1316	0	0.0000
DTC 17	0	0.00	0	0.00	1.700	0	0.0000	1	0.0592
DTC 19	2	4.51	2	4.79	2.360	1	0.3945	0	0.0000
DTC 21	2	6.38	0	0.00	3.350	0	0.0000	0	0.0000
DTC 23	0	0.00	0	0.00	4.750	0	0.0000	0	0.0000
Total	**983**	**74.67**	**436**	**34.70**	**Total**	**533**	**0.8799**	**472**	**0.3148**

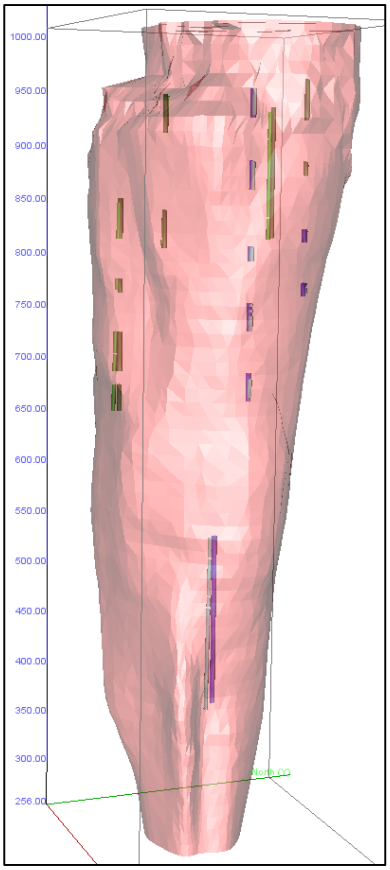

Figure 14-13: Location of LDD and adjacent microdiamond samples used as a basis for modelling total (+0.15 mm) diamond SFD curves for the M/PK(S) and EM/PK(S) domains. Dark and light blue traces are EM/PK(S) LDD and microdiamond samples, respectively. Dark and light green traces are M/PK(S) LDD and microdiamond samples, respectively. Diamond parcels recovered from these samples are shown in Table 14-16.

The diamond data shown in Table 14-16 were plotted in grade-size space (Figure 14-14) to generate models of the total diamond content SFD (larger than 0.15 mm) for the M/PK(S) and EM/PK(S) domains, respectively. Models were generated based on quadratic best-fit functions adjusted slightly to optimise the fit to the sample microdiamond and macrodiamond data. The curves in the size ranges DTC7 and larger were modelled in such a way as to exactly duplicate the +DTC7 grades of the LDD parcels on which they are based. Recovery correction factors were applied to the DTC1, DTC3 and DTC5 size classes to duplicate the +1 mm[1] recovery characteristics of the LDD diamond parcels. This provides a calibration between microdiamond stone frequency (stones per kilogram) and LDD recovered (+1 mm) grade (carats per tonne). The resultant SFD models are illustrated in Figure 14-14.

[1] Note that +1 mm grade here refers to the +DTC2 LDD sample grade to ensure consistency with estimates above 604 masl. As discussed in Section 14.4.1 the grade interpolation dataset used above 604 masl most closely approximates the LDD +2DTC sample grades. All references to +1 mm grade in Section 14.4 refer to +2DTC grade as recovered from LDD samples.

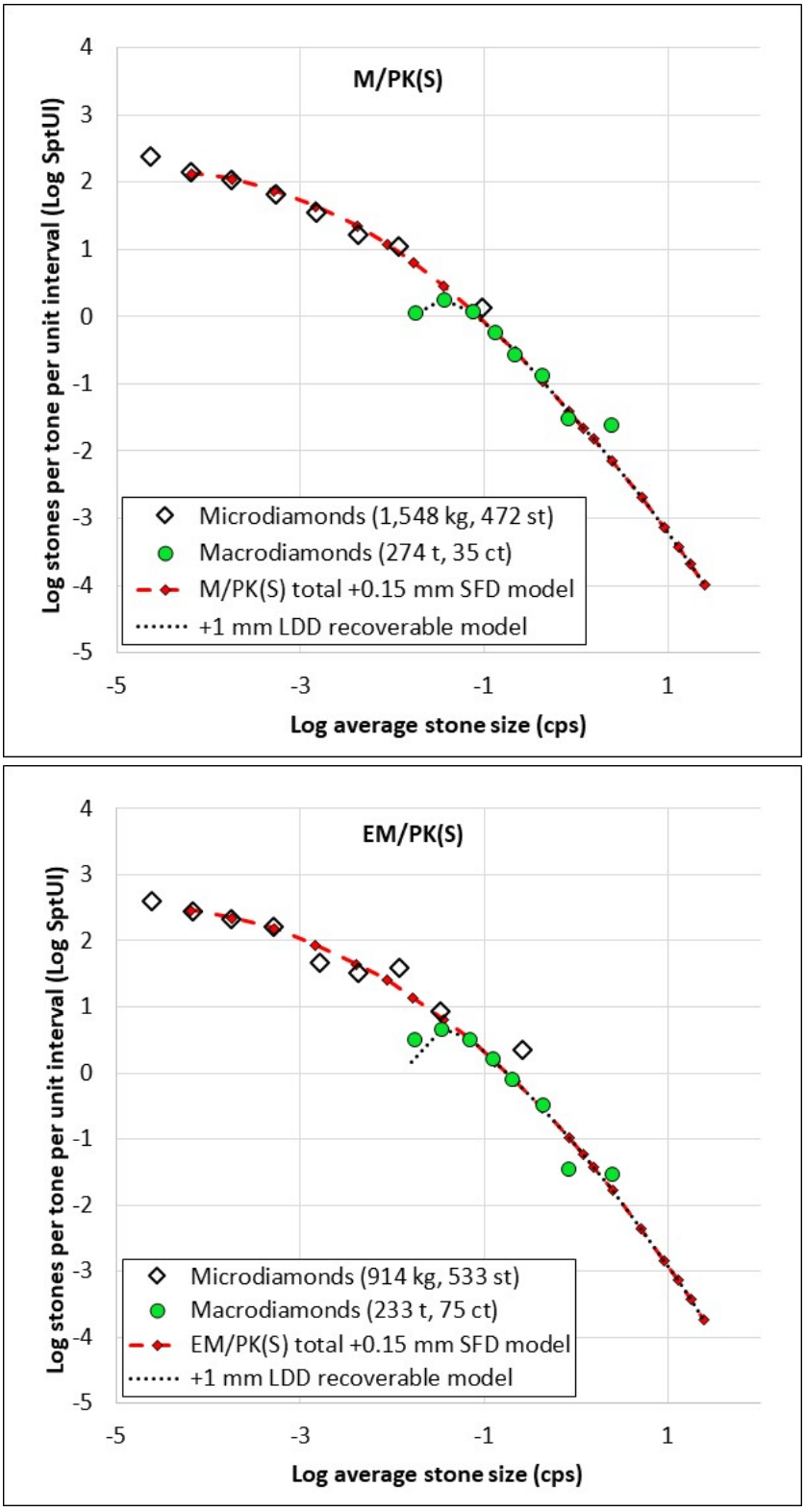

Figure 14-14: Total content (+0.15 mm) and +1 mm (LDD) recoverable SFD models for the M/PK(S) and EM/PK(S) domains.

Grade estimates

Comprehensive sampling of the 2017 deep drill holes (Section 14.4.2) provides a spatially representative microdiamond sample for the M/PK(S) and EM/PK(S) domains below 604 masl. The sample data were composited on 12 m intervals to provide a more statistically robust dataset. The composited data and the average stone frequency per 12 m elevation bench below 604 masl are illustrated in Figure 14-15 and summarised by 48 m elevation ranges in Table 14-17. While the data indicate some variability, no significant trends in stone frequency with depth are evident and observed variations are considered to be within the level of precision of the approach. The average values for the data grouped by 48 m elevation ranges display a maximum deviation of 15 % (generally less than ~10 %) from the dataset average.

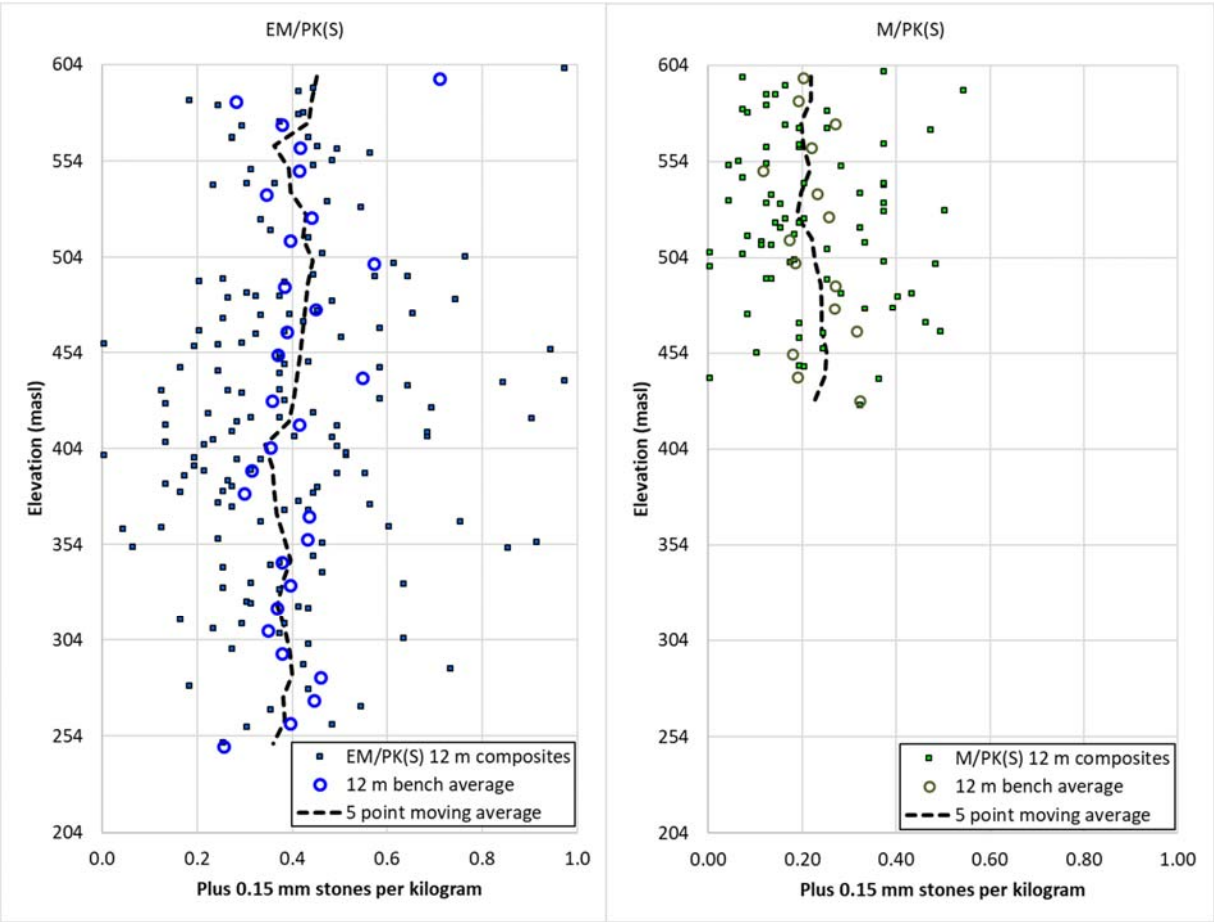

Figure 14-15: Microdiamond sample 12 m composite stone frequency results in the EM/PK(S) and M/PK(S) domains below 604 masl.

Table 14-17: Average microdiamond stone frequencies by elevation range based on composited sample data. Spkg = stones per kilogram.

Domain	Elevation range (masl) From	To	Number of composites	Average plus 0.15 mm spkg
EM/PK(S)	604	556	15	0.41
	556	508	11	0.39
	508	460	27	0.43
	460	412	33	0.41
	412	364	36	0.34
	364	316	19	0.39
	316	256	17	0.38
	Total		**158**	**0.39**
M/PK(S)	604	556	18	0.21
	556	508	29	0.20
	508	460	21	0.25
	460	412	7	0.20
	Total		**75**	**0.22**

The microdiamond sample coverage provides a broad spatial representation of the South Lobe below 604 masl, but it is not regularly distributed and does not support reliable local estimation of grade. On the basis of the observed grade and SFD continuity (Sections 14.4.3 and 14.4.4) and the lack of evidence for significant large-scale variation in microdiamond stone frequency with depth (Figure 14-15; Table 14-17) the use of average grade estimates by domain below 604 masl is considered appropriate. The bench average +0.15 mm microdiamond stone frequencies based on composited sample data were therefore used in conjunction with the defined ratios between stone frequency and grade based on total diamond content SFD models (as per above) to estimate +1 mm LDD recoverable grades of 0.15 and 0.31 cpt for M/PK(S) and EM/PK(S), respectively (Table 14-18). For consistency with the grade estimates above 604 masl these estimates were converted to a per unit volume basis using bulk density averages by bench (Section 14.3.2, Table 14-5). The calculated carat per cubic metre grades (Table 14-18) were initialized into the block model by domain and elevation range.

Table 14-18: Grade estimates for the South Lobe below 604 masl. Grade on a per unit mass basis (cpt) was calculated as per the methods explained in the text above and was converted to carats per cubic meter using bench average bulk densities.

Domain	Elevation (masl)		Bulk density	+1 mm	+1 mm Grade
	From	To	(g/cm^3)	Grade (cpt)	(cpm^3)
EM/PK(S)	604	592	2.89	0.31	0.91
	592	580	2.93		0.92
	580	568	2.96		0.93
	568	556	2.95		0.92
	556	544	2.97		0.93
	544	532	2.98		0.93
	532	520	2.99		0.94
	520	256	3.02		0.94
M/PK(S)	604	256	3.05	0.15	0.45

14.4.7 Adjustment for production plant recovery efficiency

The recovery efficiency of a production plant is by nature variable and depends on plant configuration / maintenance and ore properties; modifications to the plant process and changing physical properties of ore (e.g. increasing competency with depth) will affect the overall efficiency with which diamonds are liberated and recovered. The Karowe process plant has undergone modifications since commencement of production in 2012. The most recent upgrades (Section 13) included installation of an XRT circuit treating the 50 to 125 mm material prior to milling (to reduce breakage of large diamonds) and the installation of an additional XRT circuit to treat material in the size range 4 to 8 mm to reduce the load on the DMS. During recent controlled production test work in February 2018 the MagRoll was deactivated as it was found to have been negatively impacting fine diamond recovery. Recent production data subsequent to this have therefore been used to derive an appropriate correction to convert +1 mm LDD grades into +1.25 mm recoverable grades for the Karowe plant in its current configuration.

The controlled production run of EM/PK(S) material (Section 14.4.1) was used to compare plant- and LDD-recovered SFDs for this domain. Subsequent to the deactivation of the MagRoll there are no significant production periods during which only M/PK(S) has been processed. A production period spanning 4 to 20 March 2018 was identified as the most useful (currently available) frame of comparison. This production parcel comprised 13,562 ct of diamond recovered from 118,749 tonnes of material representing an estimated 54 % M/PK(S), 33 % EM/PK(S) and 13 % WM/PK(S).

The LDD SFD models for EM/PK(S) and M/PK(S) (Section 14.4.6) were compared to these reference production datasets in order to quantify the grade difference resulting from reduced efficiency of fine-diamond liberation and recovery in the Karowe production plant. Because the reference production batches are not spatially equivalent to the LDD samples that provide the basis for the +1 mm LDD recovered diamond SFD models, these parcels were grade-normalised such that the grade of +DTC9

diamonds matched that of the LDD parcel forming the basis for the SFD model. Differences between the grade of the normalised production batches and that of the equivalent LDD dataset primarily reflect grade differences associated with reduced diamond recovery efficiency that predominantly affects size classes DTC7 and below. For the EM/PK(S) and M/PK(S) datasets evaluated here, this analysis suggests that grade loss due to reduced diamond recovery in the production plant amounts to 30 % and 32 %, respectively. These recovery correction values are not considered to be meaningfully different from each other and a correction factor of -30 % has been used for conversion of all +1 mm LDD grade values to +1.25 mm grades recoverable with the current Karowe plant. Note that any modification to plant configuration or change in metallurgical properties in the ore will need to be accounted for in the recoverable grade (and corresponding recoverable value) estimates going forward.

14.4.8 Summary of grade estimates

Resource tonnes and recoverable (+1.25 mm) carats were extracted from the block model by lobe and bench through volumetric reporting in GEMS. Average grades (carats per tonne) are shown in Table 14-19 and grade estimates with depth in each lobe are illustrated in Figure 14-16. The grade estimates for the individual M/PK(S) and EM/PK(S) domains change significantly at the 604 masl elevation as a result of the differing approach used. Above 604 masl grade was interpolated into these domains using soft boundaries. Higher grade EM/PK(S) data points have therefore informed blocks in the M/PK(S) domain where proximal to the domain boundary, thereby increasing the average grade of the M/PK(S) domain. Similarly, lower grade M/PK(S) data points have informed blocks in the EM/PK(S) domain where proximal to the domain boundary, thereby slightly decreasing the average grade of the EM/PK(S) domain. The average grades below 604 masl have been generated for each domain independently, effectively with a hard boundary between them. The apparent increase in the grade of EM/PK(S) and decrease in grade of M/PK(S) across the 604 masl horizon reflects this differing approach. While this has a marked effect on the average grade for the domains themselves, there is no material change in the average grade of the South Lobe across the 604 masl elevation (Figure 14-16). Below 604 masl the average grade of the South Lobe increases with depth as the relative proportion of EM/PK(S) increases. The EM/PK(S) domain represents ~17 % of the pipe infill above 604 masl. Below 604 it represents 65 % of the pipe infill. This accounts for an increase of 45 % in the average grade of South Lobe from above to below the 604 masl elevation.

Table 14-19: Summary of final recoverable (+1.25 mm) grade estimates. The estimates above 604 masl are based on interpolation of LDD sample grades (Section 14.4.5). Below 604 masl grades have been estimated using a microdiamond-based approach as documented in Section 14.4.6. Mt = million tonnes, Mct = million carats, cpt = carats per tonne.

Elevation	Lobe	Tonnes (Mt)	Carats (Mct)	Grade (cpt)
Above 604 masl	North	2.69	0.49	0.18
	Centre[1]	8.79	1.49	0.17
	South	40.70	4.99	0.12
Below 604 masl	South	22.75	4.04	0.18
Total (undepleted) resource		**74.93**	**11.02**	**0.15**

[1] The Centre Lobe above 604 masl includes a small volume of material below 604 masl. The grade of this material was estimated on the basis of interpolated LDD sample results (Section 14.4.5).

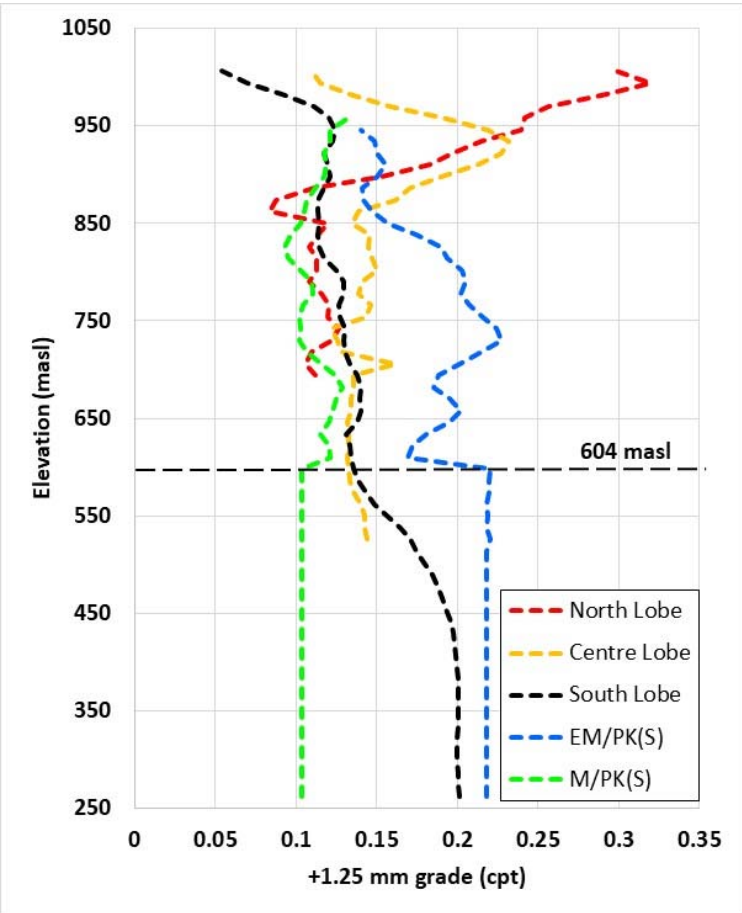

Figure 14-16: Average recoverable (+1.25 mm) grade estimates with depth in each lobe. The average grade of the South Lobe (black dotted line) does not change materially across the 604 masl elevation, confirming good consistency between the different grade estimation approaches used above and below this elevation. The increase in the South Lobe average grade with depth reflects an increasing proportion of the higher grade EM/PK(S) domain with depth. The step-change in grade for the EM/PK(S) and M/PK(S) domains at 604 masl reflects the differing grade estimation approaches used above and below this horizon, as explained in the text above.

14.5 Diamond value

The diamond value estimates in this section have been generated by Dr John Armstrong (Section 2.3) and are based on production, valuation and sales data compiled and maintained by Dr Armstrong in his role as Vice President, Mineral Resources at Lucara. MSC has reviewed the data and methods upon which these value estimates are based and considers them to be reliable.

In excess of 2 million carats of diamond produced from AK6 have been sold up to the end of Q1 2018 generating revenues of US$1.25 billion for an average price of US$606 per ct (Table 6-1). Diamond recoveries from mine production batches are sorted into DTC-Grainer-Carat[1] size classes that are typically used for valuation and sale of diamonds. The Lucara price book is applied to these sorted diamond parcels and all single diamond lots are assigned individual reserve prices based on estimated sales outcomes. This pre-sales valuation exercise produces value distribution estimates ($/ct per size class) for each mine production batch.

Lucara has carried out 44 diamond sales since inception, including 34 conventional production sales and 10 Exceptional Stone Tenders (ESTs) in which extremely large high-value diamonds are sold separately. Prior to sale the production batches available (with EST diamonds extracted) are rolled together into groupings (sales lots) of various size ranges sorted by colour and quality. Due to the grouping of diamonds from different size classes into combined sales lots it is only possible to reconcile sales data with the total average valuation estimate for each sales batch.

The diamond values in this estimate are calculated on the basis of +1.25 mm SFD models (percentage carats per sieve size class) as recovered by the Karowe production plant (and sized into valuation size classes) combined with value distribution models (US$ per carat per sieve size class). This approach ensures that the value estimates are compatible with the +1.25 mm grade estimates presented in Section 14.4.

As discussed in Section 12.6.1 the Karowe Mine maintains accurate records of the source of plant feed on a daily basis. It is therefore possible to select and group production batches derived predominantly from the North, Centre and South Lobes, respectively, providing a basis for the value estimates described in the sections below.

14.5.1 Size distribution models

The production datasets and size distribution models used as a basis for the AK6 value estimates are shown in Table 14-20. The datasets are derived from mine production batches as follows:

- North Lobe – diamond recoveries (57,252 ct) from 4 production batches during the period 2012 to 2013, with production material sourced exclusively from the North Lobe.

[1] DTC = Diamond Trading Company. DTC size classes used for valuation include DTC 3 to 11. The grainer and carat size classes are used for diamonds larger than DTC 13 size class, which are divided into size classes by mass and not by size. A carat = 0.2 g; a grainer = 0.25 carats.

- Centre Lobe – diamond recoveries (257,188 ct) from 16 production batches during the period 2012 to 2014, estimated to represent a blend of 72 % Centre, 8 % North and 20 % South Lobe kimberlite, respectively.
- South Lobe – diamond recoveries (511,435 ct) from 40 production batches during the period 2015 to 2018, estimated to comprise 93 % material derived from the South Lobe.

The selected diamond parcels derive from production periods prior to, between and subsequent to plant modifications made in 2015 and 2017 (Section 13). The parcels will therefore reflect varying process efficiency. The potential effect of this has been assessed and is considered to be minor, with a negligible effect on overall revenue estimates, and no correction has therefore been applied to account for varying process efficiency.

Table 14-20: Selected production data representing the North, Centre and South Lobes, and recoverable (+1.25 mm) SFD models derived therefrom. The data are represented in DTC, grainer (gr) and carat (ct) size classes.

Size Class	Production (ct)			Production (% ct)			SFD model (% ct)		
	North	Centre	South	North	Centre	South	North	Centre	South
+10.8 ct	579	8,836	36,024	1.01	3.44	7.04	0.95	3.05	6.42
6 - 10 ct	1,140	5,626	16,013	1.99	2.19	3.13	2.37	2.94	4.35
3 - 5 ct	3,552	14,378	30,857	6.20	5.59	6.03	5.28	3.95	5.92
8 - 10 gr	4,058	14,263	27,140	7.09	5.55	5.31	7.70	7.20	5.29
3 - 6 gr	14,732	50,292	87,628	25.73	19.55	17.13	25.73	19.43	16.99
+11 DTC	14,130	53,852	93,346	24.68	20.94	18.25	24.68	21.02	18.19
+9 DTC	9,116	41,516	78,568	15.92	16.14	15.36	15.92	15.91	15.31
+7 DTC	5,288	28,524	55,318	9.24	11.09	10.82	9.24	11.00	10.70
+5 DTC	4,584	36,214	77,643	8.01	14.08	15.18	8.01	14.20	15.10
+3 DTC	73	3,686	8,897	0.13	1.43	1.74	0.13	1.30	1.70
Total	**57,252**	**257,188**	**511,435**						

14.5.2 Value distribution models

Sales results per size class are not available due to rolling of diamonds from different size classes into sales lots. The reserve value estimates per size class (based on the Lucara price book) have therefore been used to constrain value distribution for all size classes smaller than 10.8 ct. The pre- and post-sales reports reviewed by MSC confirm that the reserve price for the -10.8 ct diamonds typically under-values these diamonds (by approximately 6 to 15 % during the period January 2015 to March 2018) relative to their average achieved sales values. The $/ct value estimate for the +10.8 ct stones is based on a combination of reserve and actual sales data from +10.8 ct lots and individual stone sales, excluding the Constellation diamond (813 ct sold for $63.11 million at US$77,649 per carat) and the Lesedi la Rona diamond (1,109 ct sold for $53 million at US$47,791 per carat). Valuation data used as a basis for value distribution modelling are shown in Table 14-21. Value distribution models (Table 14-21) were created to correct for the discrepancy between reserve and sale values, ensuring that the resulting average value estimates reconcile with overall average values achieved from the sales. Value models for the Centre and North

Lobes (based on valuation and sales results prior to 2015) have not been adjusted for recent market conditions as the modification would be negligible.

Table 14-21: Pre-sales value estimates per size class and final value distribution models for the North, Centre and South Lobes presented in DTC (Diamond Trading Company), grainer (gr) and carat (ct) size classes. These value distribution models were used in combination with the SFD models presented in Table 14-20 to generate average diamond values per lobe (Table 14-22).

Size Class	Value estimate (US$ per carat)			Value distribution model (US$ per carat)		
	North	Centre	South	North	Centre	South
+10.8 ct[1]	1,425	5,849	8,201	1,600	6,050	8,100
6 - 10 ct	1,033	1,082	1,064	1,127	1,357	1,218
3 - 5 ct	753	623	671	808	651	677
8 - 10 gr	451	406	438	484	436	445
3 - 6 gr	235	203	216	223	210	221
+11 DTC	118	95	100	95	95	102
+9 DTC	84	71	71	64	70	72
+7 DTC	63	56	49	56	56	51
+5 DTC	52	47	42	47	47	43
+3 DTC	38	49	39	35	42	39

[1] Values in the +10.8 ct size class are derived from actual sales data and not from pre-sales valuations (as for all other size classes). Large high-value diamonds from Exceptional Stone Tender sales are included. Sales results from the Constellation and Lesedi la Rona diamonds are excluded.

14.5.3 Average value estimates

The SFD and value distribution models presented in Tables 14-20 and 14-21 were combined to generate estimates of average +1.25 mm recoverable diamond value per lobe, as shown in Table 14-22. These estimates have been combined with estimates of recoverable carats in the Mineral Resource statement provided in Table 14-25. The very high value for the South Lobe in relation to the North and Centre Lobe is due to a substantially higher proportion of large diamonds with higher average values being recovered from the South Lobe.

Table 14-22: Average recoverable (+1.25 mm) diamond value estimates per lobe. Estimates are reported in US$/ct and reflect current sales values (to end of Q1 2018) for Karowe Mine diamonds.

Lobe	Average value (US$/ct)
North	222
Centre	367
South	716

14.6 Confidence and resource classification

14.6.1 Confidence in volume estimates

The pipe shell model for AK6 is constrained by 170 pierce points from 84 core and LDD drill holes. The majority of these pierce points (n = 147) fall in the upper portion of the pipe above 600 masl. In this shallower zone the shell (for all 3 lobes) is very well constrained by these pierce points and by extensive internal coverage that provides further minimum constraints on the size of the body. Fewer pierce points (n = 22) are present between 600 and 350 masl in the South Lobe[1] and in this depth range the shell is less precisely constrained. While there is scope to significantly modify the exact position of the shell in the large gaps between pierce points in this elevation range, it is highly unlikely that the overall volume could deviate by more than ±10 % from the modelled estimate. Reasons for this include:

- The high degree of confidence with which the shell is constrained above 600 masl and the good continuity with depth in the well-established side-wall dip as confirmed by deeper pierce points.
- Recent (2017) deep core drilling provides reasonable internal coverage in this elevation range that provides additional minimum constraints on the pipe volume.

Only a single pierce point is present below 350 masl (internal coverage is present to the base of the model). Below this level the shell model is predominantly based on downward continuation of established wall rock dips and there is consequently a high degree of uncertainty in the overall pipe volume.

The internal geological domain model is constrained by 18,923 m of internal core drilling. The degree of control on the boundaries between the internal domains is relatively high between surface and 450 masl. Only M/PK(S) and EM/PK(S) extend below this depth and there are no intersections of M/PK(S) below 425 masl. The available drill coverage suggests that M/PK(S) is present as a tapering feeder pipe within the EM/PK(S) domain (Figure 7-6, Section 7.4.2) and below 425 masl the relative volumes of M/PK(S) and EM/PK(S) are not constrained other than by reasonable internal drill coverage (intercepts of EM/PK(S)) confirming where M/PK(S) is not present.

14.6.2 Confidence in bulk density and tonnage estimates

Bulk density in AK6 is considered to be constrained to a high level of confidence by a large, spatially representative dataset. Local variation (maximum of ~20 %, generally less than <10 %) from the estimated bulk density is likely to be present on a small scale (e.g. on the order of a 12 by 12 by 12 m block scale) as a result of variation in dilution and alteration state, but it is unlikely that bulk density variation will result in tonnage inaccuracies on a scale pertinent to mining and resource reconciliation (i.e. on a monthly or quarterly basis).

[1] The North Lobe shell extends to a maximum depth of 690 masl. The Centre Lobe shell extends to 520 masl, but the volume of Centre Lobe present below 600 masl is not meaningful (~2 %). Discussions of geological model confidence below 600 masl are therefore focused on the South Lobe only.

14.6.3 Confidence in grade estimates above 604 masl

As indicated in Section 14.4.1 the LDD sampling provides a well-distributed spatially representative grade dataset to a depth of 604 masl, providing a basis for high confidence estimates of LDD-recoverable (+1 mm) grade per unit volume (cpm^3) above this elevation.

The +1 mm grades have been converted (through application of a 30 % downward correction) into +1.25 mm grades as recoverable by the Karowe plant in its current configuration. This was determined based on comparison of LDD and production diamond data for EM/PK(S) and M/PK(S), respectively, and primarily accounts for differences in recovery efficiency in the finer size fractions (Section 14.4.7). There are other factors that potentially influence the SFD and grades of LDD versus equivalent production parcels. Compared to production, LDD parcels would be expected to show probable higher degrees of diamond breakage, in particular affecting the coarse end of the size distribution. They may also have been impacted by a net loss of diamonds below the bottom cut-off of the process due to breakage. Finally, due to the relatively small size of the LDD samples, there will be a tendency for very large diamonds to be underrepresented in the LDD dataset, resulting in a potential slight underestimation of grade. It is not possible to quantify these potential effects based on available data and they have not been explicitly accounted for in the above-described analysis. However, they are unlikely to represent significant sources of error in the grade estimates. While diamond breakage reduces concentrations in the largest stone sizes and has a significant impact on estimation of diamond value, the broken diamonds are redistributed into the size classes below and much of the grade is preserved in the sample, thereby minimising the impact on total sample grade. Although it is not considered to be significant factor, to the extent that there is a net loss of diamonds in the LDD parcels due to broken fragments passing through the bottom cut-off screen, this would imply a slight upside on the +1.25 mm recoverable grade estimates. Similarly, the potential under-representation of very large diamonds in the LDD datasets implies minor possible upside on the estimates of +1.25 mm grade recoverable during production. The maximum extent of uncertainty associated with the calculation of the recovery correction factor cannot be quantified but is considered to be on the order of ±10 %. It must be noted, however, that any modification to the plant process or significant change in metallurgical properties (e.g. hardness) of the ore being processed may necessitate significant revisions to this correction factor.

As discussed in Section 14.4.5 soft boundaries were used for grade interpolation in the lower grade M/PK(S) and the higher grade EM/PK(S) domains. Data points within the EM/PK(S) domain have therefore informed blocks in the M/PK(S) domain, thereby slightly increasing the average grade of the M/PK(S) domain where proximal to the domain boundary. Similarly, data points within the M/PK(S) domain will have informed blocks in the EM/PK(S) domain, thereby slightly decreasing the average grade of the EM/PK(S) domain where proximal to the domain boundary. The implication of this for accuracy in grade estimates was assessed by running grade interpolations with a hard boundary. This did not result in any significant difference in terms of the overall grade estimate (~1 % difference in total carats estimated for the combined domains). The soft boundary interpolation used does under-represent the difference in

grade between these domains (Figure 14-16, above 604 masl). However, the modelled boundary between these domains, while broadly accurate and based on spatially representative drill coverage, is not precisely demarcated. Furthermore, inclusions of EM/PK(S) are present in M/PK(S) in proximity to the domain boundary (Section 7.3). The interpolation of a spatially representative and well distribution grade dataset is thought to provide the most reliable representation of the grade distribution throughout the body above 604 masl.

The volume / tonnes of kimberlite and total carats predicted by the Mineral Resource Estimate were extracted from the block model using the 31 December 2017 mine surface. The results of this are compared with the actual mined tonnes and carats produced in Table 14-23. All of this production is from well above the 604 masl elevation.

Table 14-23: Karowe Mine production to date (as of end 2017) in comparison with the equivalent resource extracted from the block model. Production and stockpile records were derived from the records used to produce Tables 6-1 and 14-25.

Source	Volume (Mm^3)	Density (tpm^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	$/ct
Resource estimate	6.37	2.45	15.61	2.23	14.3	517
Production and stockpiles	N/a	N/a	15.53 [1]	2.42 [2]	15.6	574 [3]

[1] Records of tonnes of kimberlite mined, includes plant feed and stockpiles.

[2] Total carats recovered during mining plus estimated carats in stockpiles.

[3] Calculated on the basis of total carats recovered during production (not total carats sold).

14.6.4 Confidence in grade estimates below 604 masl

The grade estimates below 604 masl are based on a calibration of microdiamond stone frequency to LDD-recovered +1 mm macrodiamond grade from selected LDD samples. Incorrect calibration of this relationship could occur if the material sampled for microdiamonds is not the same average grade as the macrodiamond sample. The datasets on which these calibrations are based are large (Table 14-16) and derive from different locations and elevations in the pipe (Figure 14-13), providing a reliable average basis for defining this relationship.

The bulk density values used to convert the mass-based grades (cpt) estimated by this method into volume-based grades (cpm^3) for inclusion with overlying (above 604 masl) grade estimates in the block model are not considered to have introduced any significant error.

Average microdiamond stone frequency values in the M/PK(S) and EM/PK(S) domains were applied to the calibrated relationship referred to above to derive average grade (cpt) estimates below 604 masl. The microdiamond datasets used to derive these averages are large (402 aliquots weighing 3.22 tonnes from the EM/PK(S) domain and 171 aliquots weighing 1.39 tonnes from the M/PK(S) domain) and broadly spatially representative, providing a reliable basis for global grade estimation. The M/PK(S) domain is not

intersected by drill core below 425 masl and continuity in grade and SFD within M/PK(S) significantly beyond this elevation cannot be assessed. The relative volume of M/PK(S) to EM/PK(S), a primary driver of average grade variation with depth, is also therefore uncertain below 425 masl, although, based on drill coverage EM/PK(S) must be the volumetrically dominant unit.

The limited LDD grade data that are available below 604 masl are presented in Table 14-24 in comparison with the average grade estimate for the elevation range represented by these LDD samples. The M/PK(S) LDD parcel below 604 masl is very limited, comprising 13.72 ct recovered from 51.8 tonnes from 2 LDD holes and does not provide a useful frame of comparison. The EM/PK(S) parcel is more substantial, including 93.04 ct from 287.3 tonnes collected from 4 LDD holes. One of these holes is LDD027, for which results are available down to 316 masl, providing a long intersection through a significant proportion of the EM/PK(S) domain below 604 masl. The +1.25 mm recovery corrected LDD grade for EM/PK(S) provides an encouraging validation of the predicted average grade for this domain.

Table 14-24: Comparison of LDD grades below 604 masl with block model grades in equivalent elevation ranges predicted by the microdiamond-based approach described in Section 14.4.6.

Domain	Elevation range (masl)	LDD tonnes	LDD carats	LDD grade (+1 mm cpt)	LDD grade (+1.25 mm cpt)	Block model grade (+1.25 mm cpt)
EM/PK(S)	604 to 316	287.3	93.04	0.32	0.24	0.22
M/PK(S)	604 to 556	51.8	13.72	0.26	0.20	0.10

The EM/PK(S) and M/PK(S) domains have been shown to display large-scale internal continuity with depth. Small scale inhomogeneity is however to be expected in kimberlite of this nature, and deviations from the predicted average grade will be present. This small scale local variability is not expected to translate into large scale inaccuracies on a level pertinent to resource reconciliation on a monthly or quarterly basis.

14.6.5 Confidence in diamond value estimates

The SFD models and value distribution models from which average diamond values are estimated (Section 14.5) are based on the results of substantial actual diamond production and sales. MSC has reviewed the relevant data sources and the calculation of these average values and considers them to be constrained to a high level of confidence.

The average values, based on results of near-surface production (from surface to ~900 masl) have been adopted by lobe from surface to the base of the Mineral Resource Estimate at 256 masl. The projection of constant diamond value with depth is based on an assumption of geological and diamond SFD continuity with depth within each lobe. Geological continuity and diamond SFD characteristics have been extensively investigated as described in Sections 14.2.2, 14.4.3 and 14.4.4. It is not possible to quantify an associated level of uncertainty, but the authors consider the assumption of SFD constancy with depth to be valid and to have been demonstrated to a degree of confidence adequate for the declaration of Indicated Mineral Resources.

The diamond value estimate for the South Lobe is based on production data predominantly derived from the M/PK(S) domain and its weathered / diluted equivalents. A key area of risk in diamond value estimates is the possibility for the EM/PK(S) domain, while presenting similar SFD characteristics in the LDD data available, to manifest a different SFD and potentially not contain the same proportion of large very high value diamonds that underpin the high average value estimate for the South Lobe. The results of the large controlled production run from the EM/PK(S) domain (Section 14.4.1) provide compelling evidence to mitigate this risk. Key results include:

- The sample returned 9.5 weight % carats of diamond larger than 10.8 ct;
- The valuation report for the EM/PK(S) diamonds (GTD Consulting, 2018) documents an average value of US$753 per ct;
- More than 83 % of the total value derives from the +10.8 ct size fraction (n = 47), which includes 1 diamond larger than 100 ct, 6 diamonds larger than 50 ct and 3 diamonds larger than 30 ct;
- The highest value diamond was a 72.84 ct stone valued US$60,000 per carat;
- Five additional diamonds valued in excess of US$10,000 per carat were present;
- The average value of the +10.8 ct diamonds was estimated at US$7,058 per carat.

Results from this controlled production run provide confirmation, therefore, that the EM/PK(S) diamond population has an exceptionally coarse-grained SFD with high proportions of large very high value diamonds, equivalent to the well-established characteristics of the diamond population derived from the M/PK(S) domain. The confidence in the average values adopted is considered to be adequate for the declaration of Indicated Mineral Resources.

Table 14-24 in Section 14.5.4 shows a reconciliation of tonnes, carats and grade mined to date with the Mineral Resource Estimate. The table also includes a reconciliation of diamond value results from sales with the corresponding average value extracted from the Mineral Resource Estimate. The minor discrepancy reflects the exclusion of the very high value Constellation and Lesedi la Rona diamonds (~9 % of mine revenue to date) from the value estimates (Section 14.5).

14.6.6 Resource classification

All components (volume, tonnage, grade and value) of this estimate from surface to an elevation of 400 masl are considered to be constrained to a level of confidence suitable for the classification of Indicated Mineral Resources. Note that confidence in grade, primarily driven by uncertainty in the relative volume of M/PK(S) and EM/PK(S), decreases in the lower portion of this elevation range (400 to 450 masl, as discussed in Sections 14.6.1 and 14.6.4). The implication of increased uncertainty in this deeper material is limited within the context of the overall Indicated Mineral Resource reported from surface to 400 masl, but future assessments and mine planning should take this into account.

From 400 to 256 masl (the base of the geological model) the confidence in volume and grade is lower. In this elevation range the estimate is considered to be constrained to a level of confidence suitable for the reporting of Inferred Mineral Resources.

14.7 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 14-25. Estimated tonnes and carats reflect the depleted resource, with material mined up to the end of December 2017 removed from the original model. Resource grade and average value estimates (updated from those reported in Oberholzer et al., 2017) reflect expected recoverable diamond production using the current 2018 Karowe plant configuration with a bottom cut-off of 1.25 mm. 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 2017 and do not account for subsequent mining depletion. For reasons outlined in the sections above, the upper ~ 600 m of the deposit (to an elevation of 400 masl) has been classified as an Indicated Mineral Resource, comprising an estimated total of 53.48 million tonnes of kimberlite ore, containing 7.62 million carats of diamonds at an average diamond value of $674 per carat. Stockpiles at the Karowe Mine as of 31 December 2017 were estimated to contain 2.33 million tonnes of kimberlite containing 0.29 million carats of diamonds at an average diamond value of $645 per carat (based on stockpile inventories maintained by Karowe Mine). Mineral Resources contained within the Karowe stockpiles are not constrained at confidence levels typically required for independent classification as an Indicated Mineral Resource. However, this material makes up less than 4 % of the total resource above 600 masl, substantially mitigating the potential impact of this uncertainty. The stockpile Mineral Resources have therefore been accumulated with Indicated Resources in Table 14-25.

The portion of the deposit from 400 masl to the base of the model at 256 masl is classified as an Inferred Mineral Resource, with an estimated total of 5.84 million tonnes of kimberlite ore, containing 1.17 million carats of diamonds at an average diamond value of $716 per carat

Table 14-25: Statement of the estimated remaining Mineral Resource in the AK6 kimberlite. Resources are those remaining (including stockpiles) at end December 2017. LOM = life of mine, SP = stockpile, Mm^3 = million cubic metres, tpm^3 = tonnes per cubic metre, Mt = million tonnes, cpt = recoverable (+1.25 mm) carats per tonne, Mct = million carats, $/ct = recoverable (+1.25 mm) United States dollars per carat).

Classification	Resource	Volume (Mm^3)	Density (tpm^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	$/ct
Indicated	North Lobe	0.62	2.48	1.54	0.20	13.0	222
	Centre Lobe	1.68	2.57	4.32	0.63	14.6	367
	South Lobe	16.29	2.92	47.63	6.78	14.2	716
	Total	18.59	2.88	53.48	7.62	14.2	674
	LOM SP	1.28	1.85	2.36	0.09	3.8	609
	Working SP	1.05	1.91	2.01	0.20	9.7	661
	Total Stockpile	2.33	1.88	4.37	0.29	6.5	645
	Total Indicated	**20.92**	**2.77**	**57.85**	**7.90**	**13.7**	**673**
Inferred	**South Lobe**	**1.93**	**3.02**	**5.84**	**1.17**	**20.0**	**716**

15. Mineral Reserves

This section was contributed by Lucara Diamond Corp. under the oversight of Henk Fourie of Lofty Mining (Pty) Ltd. (QP responsible for the Mineral Reserve Estimate). This section provides Mineral Reserve Estimates for the open pit portion of the Karowe Mine (as documented in Oberholzer et al., 2017) updated to reflect the updated Mineral Resource Estimate reported in Section 14.

Mineral Reserve estimation is based on the updated Indicated Mineral Resource Estimate. Inferred Resources have not been used to estimate Mineral Reserves. The Resource to Reserve conversion was performed by Lucara by conducting an open pit optimisation, using Whittle® suite software. The outputs of this process include a mining schedule on which to base plant capacity, waste rock quantities, peak capacities and mining fleet parameters. The mining plan is reviewed in Section 16. It should be noted that the Whittle® optimisation is ongoing and is being considered within the feasibility study of the Karowe Underground Project.

The Mineral Reserve Estimate has been classified and reported in accordance with the Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral projects' of June 2011 (the Instrument), updated in 2015 and the classifications adopted by the CIM Council in November 2011.

The effective date of the Mineral Reserve Estimate is May 2018.

The Mineral Reserves were derived from the Mineral Resource block model that is presented in Section 14.1. The Mineral Reserves are the Indicated Mineral Resources that have been identified as being economically extractable and incorporate mining losses and the addition of waste dilution. The Mineral Reserves form the basis for the mine plan presented in Section 16.

15.1 Key assumptions

Diamond recovery factors have been factored into the Mineral Resource Estimate on the basis of the current plant configuration and additional data in comparison to the 2013 estimate (Lynn et al., 2014), and have therefore not been re-factored in the estimation of the Mineral Reserve.

There are no specific grade control programs undertaken at Karowe. Generally, all ore within the resource models is considered to be economic, and is either processed directly or stockpiled for possible future processing. Mining recovery of 97 % and dilution of 3 % were applied in the optimisation to better simulate the physical operation. Plant recovery was set at 100 %.

The QP carried out a review of the open pit optimisation undertaken by Lucara. In the QP's opinion, the results of this review show that the current LOM design and proposed LOM schedule are sufficiently practical and represent the optimal pit-shell.

The process to develop the open pit Mineral Reserves for the Karowe Diamond Mine is detailed in Section 16 of this report. The key assumptions for the conversion of Mineral Resources to Mineral Reserves are described below:

1. The updated Mineral Resource Estimate detailed in this report forms the basis of the open pit optimisation.
2. Indicated Resources extend significantly beyond the limits of the open pit optimisation, hence the open pit optimisation has been undertaken exclusively in the Indicated Resources.
3. The grades and tonnes of the Mineral Resource model have been modified by a mining recovery of 97 % to allow for cross hauling and ore loss. The mining dilution is based on ore body geometry and mining methodology. A static 3 % dilution at 0.0 cpht was used to allow for waste rock inclusion into the ore blast blocks.
4. The Whittle® suite of optimisation software was used to perform the pit optimisations. Whittle® is an accepted industry optimisation tool that uses the 3D Lerchs-Grossmann algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block. The selected pit design supporting the Mineral Reserve Estimates extends to an elevation of 695 masl.
5. Diamond prices were derived from the Lucara Price Book based on historical sales and production: South Lobe – US$ 716/ct, Central Lobe – US$ 367/ct, North Lobe – US$ 222/ct.
6. A government royalty of 10 % and a Marketing cost of 1.9 % of diamond sales revenue.
7. Updated geotechnical recommendations (Terbrugge and Mossop, 2017) to maintain pit slope stability were used in the optimisation. The Overall Slope Angles (OSA) used in the pit optimisation process are described in Section 16.3.10.
8. Plant recovery of 100 % has been used in the optimisation. A 70 % modifying factor has been applied to the in-situ diamond grade at 1.00 mm to account for production recovery in the current processing plant at a bottom cut-off screen (BCOS) of 1.25 mm (Section 14.4.7). This is factored into the Mineral Resource Estimate and hence no additional plant recovery adjustment was required for conversion to Mineral Reserve Estimates.
9. Processing plant design throughput of 2.6 Mtpa.
10. Mining costs are based on the current mining contractor operating at Karowe. The base date for the mining costs is Q2 2018. Reference mining cost of US$2.65/t and a fixed monthly management fee of US$340,000 per month.
11. Processing costs are based on the current processing contractor operating at Karowe. The base date for the processing costs is Q2 2018. Processing cost of US$10.62/t and a fixed monthly General and Administration cost of US$4.25/t milled based on Karowe 2018 Budget.
12. The Mineral Reserve for the Karowe Diamond Mine was evaluated against the current pit design and is within 11 % of ore and 2 % of waste from the optimal pit shell generated by the Whittle ® open pit optimisation software.

15.2 Mineral Reserve statement

The Mineral Reserves for the open pit portion of the Karowe Diamond Mine (Table 15-1) were converted from the Indicated Mineral Resources using the modifying factors discussed in Section 16. All of the Mineral Reserve is classified as Probable based on a Resource Classification of Indicated (Section 14.7). Inferred Mineral Resources have been excluded from the conversion of Resources to Reserves.

Table 15-1: Open pit Mineral Reserve statement for the Karowe Diamond Mine.

Open Pit Mineral Reserve Estimate for the Karowe Diamond Mine, Botswana, as at May 2018						
Lobe	Reserve Category	Tonnes (Mt)	Recoverable Grade[13] (cpht)	Recoverable Carats (Mcts)	Diamond Revenue[6] (US$/ct)	Unit Revenue (US$/t)
North	Probable	1.04	13.37	0.14	222	29.68
Centre	Probable	3.37	14.57	0.49	367	53.46
South	Probable	15.43	12.74	1.97	716	91.22
In-situ Reserve (OP Material)[14]		**19.84**	**13.08**	**2.60**	**624**	**81.58**
Working Stockpiles[11]	Probable	2.10	9.96	0.21	661	65.83
LOM Stockpiles[12]	Probable	3.46	4.57	0.16	609	27.84
Total Reserve[15,16]		**25.40**	**11.66**	**2.96**	**625**	**72.95**

Notes:

1. The Mineral Reserve has been depleted for mining up to May 2018
2. Figures have been rounded to the appropriate level of precision for reporting
3. Due to rounding, some columns or rows may not compute exactly as shown
4. The Mineral Reserves are stated as in-situ dry metric tonnes
5. The Mineral Reserves were prepared under the guidelines of the CIM, for reporting under NI 43-101
6. Diamond price is based on diamonds recoverable with current Karowe plant process and Lucara Diamond Price Book
7. Modifying factors for mining recovery of 97 % and waste dilution of 3 % at 0.0 cpht have been applied
8. Probable Mineral Reserves were derived from Indicated Mineral Resources
9. Mineral Reserves are inclusive of Mineral Resources
10. There are no known legal, political, environmental, or other risks that could materially affect the potential Mineral Reserves
11. Working stockpiles comprise surface loose stocks of material with estimated grades exceeding 7 cpht; includes High Grade (HG), Medium Grade (MG), Low Grade (LG) and Contact kimberlite
12. Includes existing LOM Stockpiles of Very Low Grade (VLG) kimberlite material (< 7cpht) as well as in-situ VLG material (currently part of in-situ resource) expected to be directed to the LOM stockpile (1.0Mt @ 6.24 cpht in-situ and 2.5Mt @ 3.9 cpht current surface stocks @ average value of US$ 609/ct). LOM Stockpiles will be processed at the end of life of open pit mining
13. Based on the updated Mineral Resource estimate as presented in this report (1.25 mm bottom cut off size - BCOS) – 70 % of in-situ carats at 1.00 mm BCOS
14. Exclusive of current stockpiles and VLG in-situ material (see note 12 above)
15. Inclusive of current stockpiles and VLG in-situ material (see note 12 above)
16. The Mineral Reserves reported in this table are attributable solely to the ore to be mined (and processed or stockpiled for later processing) from the open pit mine at Karowe

16. Mining methods

This section was contributed by Lucara Diamond Corp. Sections 16.1 and 16.2 are summarised from Oberholzer et al. (2017) and have been prepared under the oversight of Dr John Armstrong. Sections 16.3 and 16.4 are based on the studies undertaken for Oberholzer et al. (2017) updated by Henk Fourie to incorporate the new Mineral Resource Estimate presented in this report along with updated costs and depletion surfaces.

16.1 Geotechnical

The Information in this section was extracted and summarized from Oberholzer et al. (2017).

16.1.1 Data sources and previous studies

Several historical geological and geotechnical reports were made available from which to extract relevant data and gain initial understanding. These included:

- Barnett (2007) provides details on the geological 3D model still in effect at the time of this report. 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 this report.
- 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 10-2. Seven of the 15 holes had triple tube core recovery suitable for geotechnical investigations.

16.1.2 3D geological model

The 3D country rock geological model after Barnett (2007) is shown in Figure 16-1 with additional detail in Table 16-1. Eight (8) country rock units (Table 16-1) were differentiated along with three (3) kimberlite lobes.

Table 16-1: Basic 3D country rock model geological units (after Barnett, 2007).

Formation	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

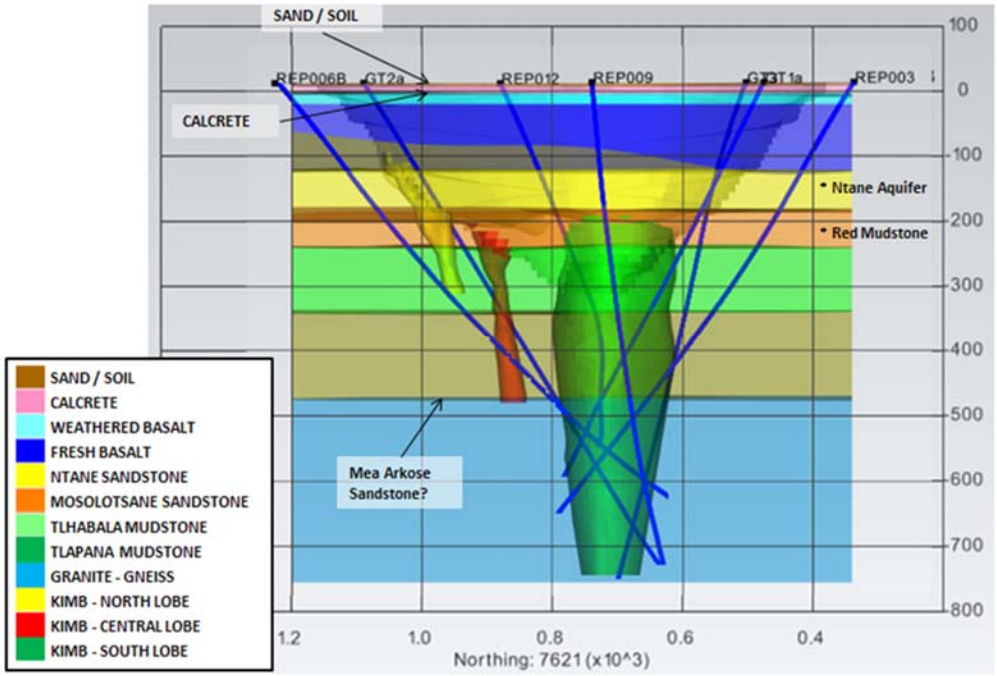

Figure 16-1: 3D country rock geological model after Barnett (2007).

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

This model, shown in Figure 16-1, has not been validated with in-pit observations. The model is currently being revised as a portion of a feasibility study for a potential underground mine at Karowe.

16.2 Hydrogeology

This section is summarised and condensed from Oberholzer et al. (2017).

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).

16.2.1.1 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.

16.2.2 De-watering of current open pit

As of November 2017, Karowe Diamond Mine operates fifteen pit perimeter dewatering boreholes; twelve electric powered and three diesel (Figure 16-2). 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.

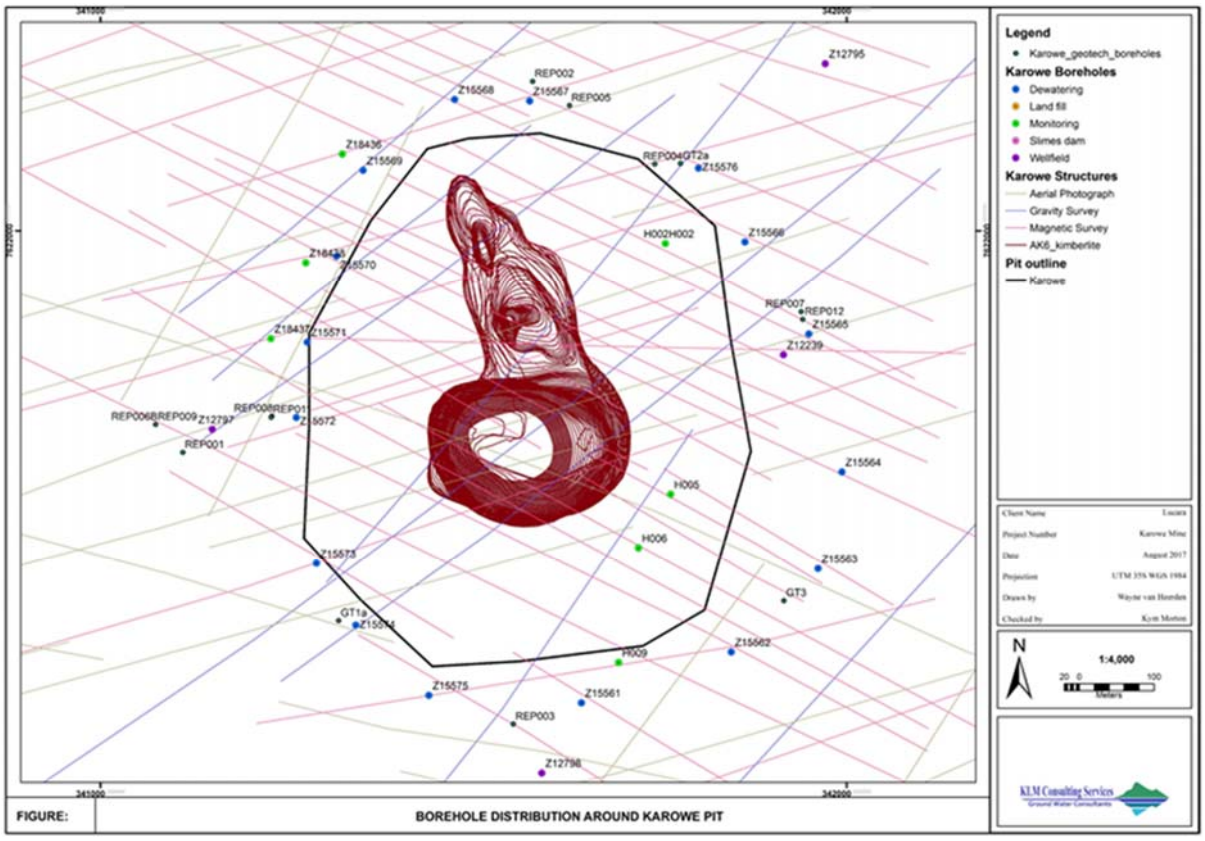

Figure 16-2: November 2017 pit dewatering boreholes and infrastructure.

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 (Itasca, 2015), the water levels are expected to be decreased to approximately 800 masl by 2021. The required dewatering rate is 25 m/yr therefore Karowe Mine has embarked on a fast tracked strategy to achieve the required dewatering rate.

16.3 Open pit mining

16.3.1 Mining method

The method of mining for Karowe Mine is a conventional open pit method using drilling and blasting, loading with excavators, and hauling with articulated dump trucks and rigid frame dump trucks. Ore and waste will be extracted by hydraulic excavators (100 to 120 t Class) and loaded into diesel off-road haul trucks (90 to 100 t Class) for discharge at the ROM crushing facility, stockpiling area or waste dump area. The mining operation is supported by ancillary equipment including bowsers, grader, dozers and front-end loaders.

The planned scale of mining at the Karowe Diamond Mine is medium scale with a 2018 peak total material movement of 15-17 Mtpa. The required mining rate will decrease as waste stripping in Cut 2 diminishes and approximately 10 Mtpa mining rate will be required for 2019 and 2020, then reducing to approximately 6.5 Mtpa until end of life of the open pit operation. The annual processing plant feed requirement is approximately 2.6 Mtpa until end of life of mine.

16.3.2 Geological block model used in pit optimisation

The three-dimensional block model (AK6 2018 Block Folder BLK exports.zip) was received from MSC on 4th June 2018. This volume percent block model consisted of 14 block folders exported from the GEMS software. Each block folder export file contained the following fields:

- Rockcode – specific numerical value representing kimberlite facie;
- Density – density of the rock in tonnes per cubic metre;
- Percent – proportion of the rock-type contained within the cell volume;
- CPM3 – Carats per cubic metre – converted to CPHT with the application of the density field;
- RCPT – recoverable carats per tonne based on 1.25 mm bottom screen cut-off size.

All the blocks had the following dimensions:

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

In addition to the GEMS model export MSC provided updated geological model (AK6_2018.dxf) containing 45 discrete model solids.

A Datamine resource block model was generated from the supplied GEMS model export and the DXF solids. The following process was used:

1. Import 14 individual GEMS block model information;
2. Import kimberlite model solids individually;
3. Fill solids on GEMS model framework with sub-celled model;
4. Join the GEMS block model table with sub-celled model on IJK key field;
5. Report original statistics and sub-celled statistics to validate;
6. Combine all kimberlite model solids to form a single geological resource model.

The outcome of the validation between the original model and the Datamine sub-celled model was 0.004 % variance on mass and 0.014 % on diamond content. This is considered immaterial and the conversion process is considered to be successfully validated. The model contents are shown in Table 16-2.

Table 16-2: Block model contents.

Folder	Description	Rock code	Ore volume (m^3)	Ore tonnage (t)	Density (t/m^3)	In-situ carats +1.00 mm	Recoverable carats (1.25 mm)	Recoverable grade (cpht)
S_INTSWB	South internal basalt raft	101	47,383	111,409	2.35	-	-	-
S_Wx	South weathered	102	1,996,315	4,590,089	2.3	764,723	535,306	11.66
S_WXBBX	South weathered country rock breccia	103	451,031	1,001,801	2.22	94,935	66,454	6.63
S_BBX	South country rock breccia	104	127,496	339,596	2.66	22,148	15,503	4.57
S_MPK	South M/PK(S)	105	12,141,916	35,588,275	2.93	5,562,805	3,893,963	10.94
S_EMPK	South EM/PK(S)	106	7,466,564	21,813,096	2.92	6,461,284	4,522,899	20.73
C_Wx	Centre weathered	201	907,796	2,002,945	2.21	465,122	325,586	16.26
C_WXBBX	Centre weathered country rock breccia	202	7,956	18,957	2.38	6,610	4,627	24.41
C_BBX	Centre country rock breccia	203	348,581	892,058	2.56	139,285	97,500	10.93
C_KIMB	Centre kimberlite	204	2,276,096	5,874,771	2.58	1,519,340	1,063,538	18.1
N_Wx	North weathered	301	352,708	798,762	2.26	305,903	214,132	26.81
N_WXBBX	North weathered country rock breccia	302	53,496	126,215	2.36	46,433	32,503	25.75
N_BBX	North country rock breccia	303	287,093	729,964	2.54	112,387	78,671	10.78
N_KIMB	North kimberlite	304	425,507	1,043,560	2.45	242,490	169,743	16.27
Total			**26,889,937**	**74,931,497**	**2.79**	**15,743,465**	**11,020,426**	**14.71**

16.3.3 Engineering block model

In preparation for the open pit analysis an engineering model was developed using the following methodology:

1. Waste country rock model was generated from an updated waste rock modelling exercise concluded by SRK (Q1 2018) for the purpose of geotechnical modelling for the underground feasibility study. In addition to the solid models supplied, SRK provided updated densities for the individual country rock strata;
2. Ore and waste models were added together;
3. Combined model was depleted using the May 2018 surveyed pit faces;
4. Model was coded for geotechnical slope considerations; and

5. Mining cost adjustment factors were calculated to adjust for an increase in mining cost as pit deepens.

The engineering model was imported into the open pit optimisation software and validated upon import. No errors during the import process were detected.

16.3.4 Mining depletion surface

Both the ore and waste models were depleted using a May 2018 pit surveyed face position DTM supplied by Karowe Diamond Mine Technical Services department (MAY18_1.5mFILT.dxf). A wireframe of the surface of the May 2018 pit position is illustrated below (Figure 16-3).

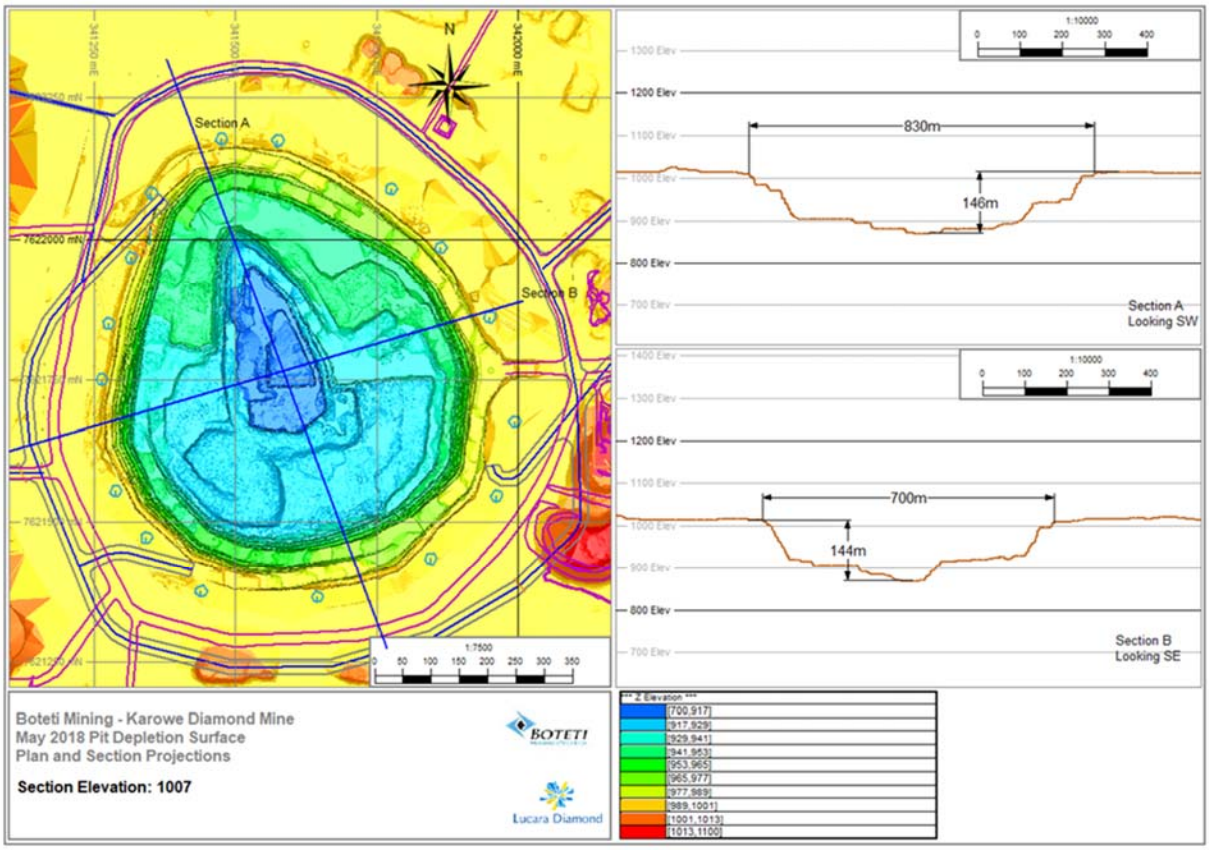

Figure 16-3: Surface topography, May 2018.

16.3.5 Exchange rate

An exchange rate of 10.25 BWP to 1 US$ was used in the optimisation and was based on 1-year historical average.

16.3.6 Mining production target and processing rate

The processing tonnage limit was guided by design capacity of the upgraded Phase 2&3 processing plant in conjunction with planned maintenance and down-time roster and set at 2.6 million tonnes per annum (Mtpa).

Based on currently detailed scheduling, mining fleet capability is currently 15 – 17.5 Mtpa and is expected to decrease to 10 Mtpa for 2019 and 2020 and 6.5 Mtpa until end of life of the open pit mining operation. The mining limit was set at 17.5 Mtpa however, the open pit optimisation schedule closely mimics the detailed production planning schedule.

16.3.7 Mining dilution and recovery

A mining dilution of 3 % was applied to allow for the mixing of waste with the kimberlite ore, typically at the contact zones. A mining recovery of 97 % was used in the pit optimisation to allow for a loss of 3 % kimberlite ore by incorrect loading and hauling of the ore to the waste dump. These factors are considered satisfactory and are aligned with the operational experience of the current operating mine.

16.3.8 Mining cost

The mining costs were derived from 2018 budget numbers based on existing contractor miner operating at Karowe Diamond Mine. The mining costs comprise:

- a reference load and haul cost of Pula (BWP) 25.63 per bank cubic metre (bcm) and BWP 24.51 per bcm for waste and kimberlite respectively;
- a drill and blast cost of BWP 12.08 per bcm and BWP 16.28 per bcm for waste and kimberlite respectively (including pre-splits in final walls);
- ancillary and support equipment at BWP 20.00 per bcm;
- diesel consumption cost of BWP 12.94 per bcm;
- a contractor monthly management fee of BWP 3.48 million per month;
- an additional BWP 3.28 per tonne has been applied to the processing cost to allow for the ore incremental cost (ore re-handle, blast pattern differences, etc.);
- waste - load and haul incremental cost, to allow for longer hauling distances as the pit deepens, was set at BWP 0.15 per bcm for volumes mined below 1,013 masl; and
- kimberlite - load and haul incremental cost, to allow for longer hauling distances as the pit deepens, was set at BWP 0.15 per bcm for volumes mined below 1,013 masl.

The reference mining cost (waste at 1,013 masl) is BWP 27.17 per tonne mined (US$ 2.65 per tonne mined at an exchange rate of 10.25 BWP per 1 US$). This increases to US$ 2.88 per tonne mined with the inclusion of monthly management fees and a mining capacity of 17.8 Mtpa.

16.3.9 Diamond prices

The diamond prices were supplied by Lucara Diamonds as reported in Section 14.5.3. The Lucara Diamond prices used in the optimisations were:

- South Lobe – US$ 716 / ct
- Central Lobe – US$ 367 / ct
- North Lobe – US$ 222 / ct

Although the existing surface stockpile did not form part of the optimisation process, Lucara Diamonds supplied the following diamond prices for existing surface stocks:

- Working Stockpiles (LG, MG, HG and contact) – US$ 661 / ct
- LOM Stockpiles (< 7 cpht) – US$ 609 / ct

No real diamond price escalations were used in the open pit optimisation analysis.

16.3.10 Geotechnical design parameters

Geotechnical slope design parameters (Figure 16-4 through Figure 16-6) were aligned with the updated geotechnical recommendations set out by SRK (Terbrugge and Mossop, 2017).

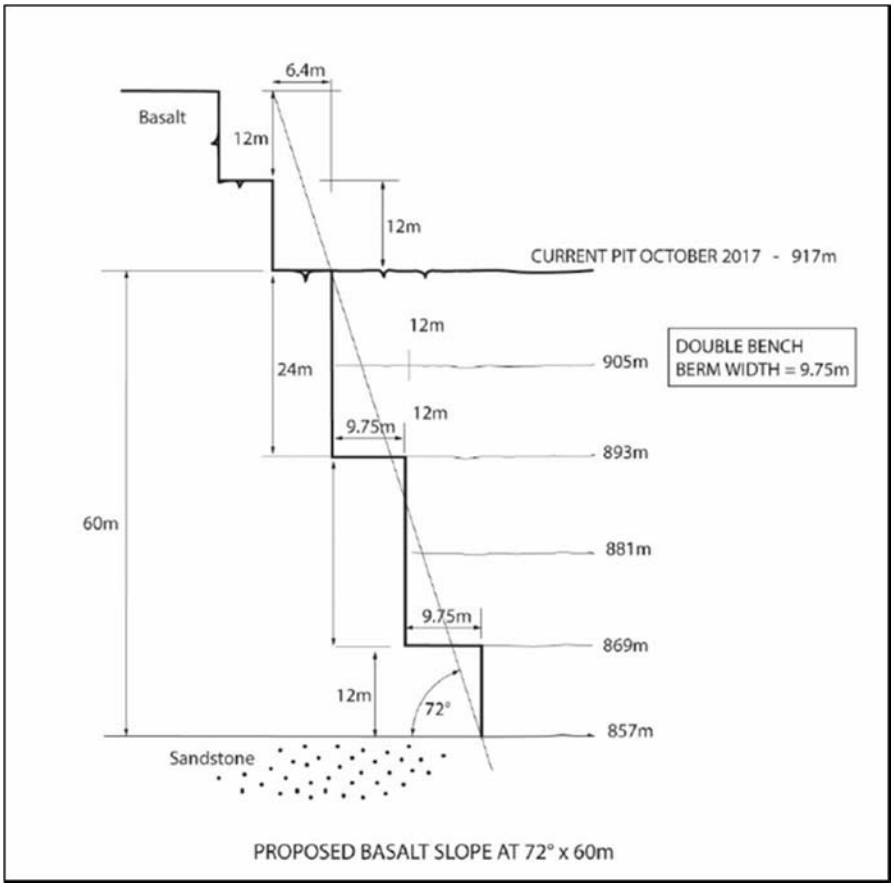

Figure 16-4: Fresh basalt slope recommendations for double-benching (Terbrugge and Mossop, 2017).

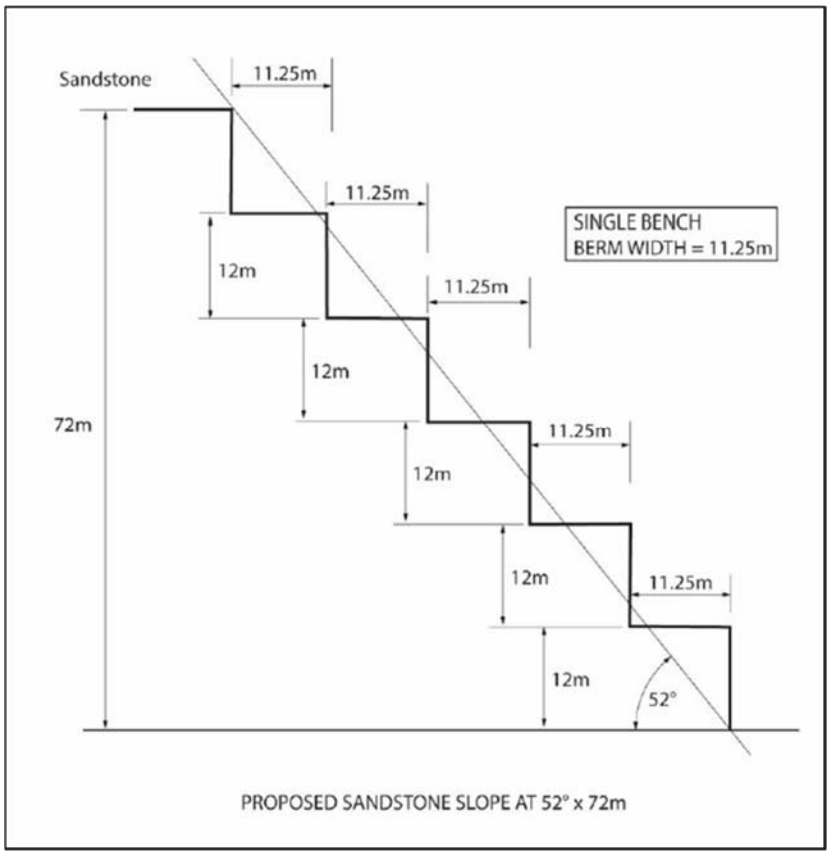

Figure 16-5: Proposed sandstone slopes (Terbrugge and Mossop, 2017).

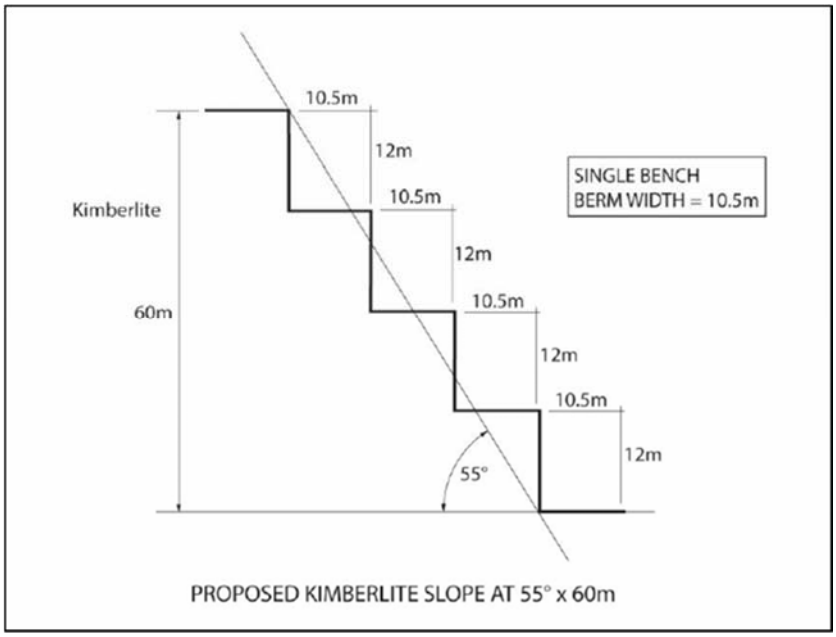

Figure 16-6: Proposed kimberlite slopes (Terbrugge and Mossop, 2017).

SRK updated geotechnical recommendations for benching of kimberlite and sandstone rock strata, as well as double-benching of the fresh basalt, were supplemented with the other country rock strata from previous study work performed by SRK in order to complete the engineering model used in the open pit optimisation. The optimisation overall slope angles used in the pit optimisation are detailed in Table 16-3 below.

Table 16-3: Geotechnical overall slope angles used in the open pit optimisation.

Parameter	Calcrete	Weathered basalt	Fresh basalt	Sandstone	Mudstone	Kimberlite
Overall slope angle for all walls	51	44.5	67.9	46.95	33.4	55.6

16.3.11 Bottom cut-off screen size

The bottom cut-off screen size (BCOS) selection was guided by Karowe Mine. A BCOS of 1.25 mm was used for all open pit optimisations. Adjustment for BCOS from 1.00 mm (resource) to plant BCOS of 1.25 mm was 70 % as detailed in Section 14.4.7.

16.3.12 Process recovery

SFD analysis (Section 14.4.7) presented in this report has informed the use of 70 % from 1.00 mm resource grade to a recoverable grade at 1.25 mm BCOS. The recoverable grades (70 % of 1.00 mm in-situ grade) has been used in the optimisation and hence the processing recovery factor was set to unity (100 %) in the pit optimisations.

16.3.13 Processing cost

The processing costs for the Phase 2 (2.6 Mtpa) Crush-Milling-DMS-diamond sorting processing plant is based on the 2018 Budget costs supplied by Karowe Diamond Mine. The cost of processing applied in the pit optimisations is BWP 108.88 per tonne milled or US$ 10.62 per tonne milled.

For the purposes of the pit optimisation, all fixed cost components are added to the processing cost and include:

- ore incremental cost;
- contractor monthly managements fee;
- general and administrative monthly costs; and
- off-mine fixed costs.

The total cost of processing applied in the pit optimisations (with the additional of the cost listed above) is BWP 171.80 per tonne milled or US$ 16.76 per tonne milled.

16.3.14 On-mine additional costs

Additional costs for general and administrative costs was aligned with the 2018 Budget supplied by Karowe Diamond Mine. The G&A costs used in the open pit optimisations was BWP 9.44 million per month (BWP 43.58 per tonne milled and US$ 4.25 per tonne milled).

16.3.15 Off-mine additional costs

No additional off-mine costs formed part of the 2018 Budget as these costs have been allocated to on-mine costs centres.

16.3.16 Selling cost and royalty

The government of Botswana legislates a 10 % royalty for precious stones calculated on diamond sales revenue.

Marketing costs were aligned with previous LOM optimisations and confirmed by Lucara Diamonds. The marketing costs were 1.9 % of diamond sales revenue.

16.3.17 Discount rate

An 8 % discount rate has been applied in the optimisations for the purposes of calculating discounted future cashflows. This discount rate has been aligned with the Lucara Diamonds financial evaluation policies.

16.3.18 Constraints

Constraints in the open pit optimisation typically consist of mining, processing and element selling limits. The following constraints / limits were used in the various open pit optimisations and sensitivities (Table 16-4).

Table 16-4: Constraint considerations in open pit optimisation.

Constraint / Target	Units	2018	2019/2020	2021	2022 – EOL
Total mined tonnage	Mtpa	17.5	10	6.5	6.5
Ore tonnage	Mtpa	2.8	2.8	2.8	4.0[Note1]
Processing throughput	Mtpa	2.6	2.6	2.6	2.6
Element limit	ct / yr	Nil	Nil	Nil	Nil

Note 1: Ore mining limit increased to allow for open pit operation to cease in 2024 considering the potential for underground blasting activities to begin in 2025, subject to positive Feasibility study and UG mine construction.

No geographical constrained areas were identified surrounding the AK6 deposit that required the optimisation to be constrained due to various surface constraints. No areas were excluded from the pit optimisation by the application of heavy-blocks or excessive mining cost adjustment factors.

16.3.19 Optimisation method

The Milwa® Balanced Algorithm was used in all optimisations. The Milwa® Balanced Algorithm seeks to optimise both the discounted pit value whilst maintaining processing throughputs, as well as adjusting the mining rate, as far as possible to the targeted mining rate / limit.

16.3.20 Life of mine pit schedule assumptions

Mining is planned to start June 2018 (model depleted to May 2018 survey face. Waste stripping in 2018 requires the current fleet to maintain an instantaneous mining rate of 17.5 Mtpa. In 2019 the waste stripping requirements reduces the mining instantaneous production rate to approximately 10 Mtpa for the 2019 and 2020 calendar years. Thereafter the peak mining fleet tonnage requirements will be approximately 6.5 Mtpa until the end of the open pit life.

16.3.21 Open pit optimisation

Run 0 (Base Case Optimisation Parameters) formed the basis of the verification of the current Cut 2 mine design and schedule for the open pit LOM study. The selected option is inclusive of Indicated Resources only, the purpose of this optimisation is to provide a key mechanism in determining the open pit potential of the deposit.

Pit-shell 38 (revenue factor 0.67 pit-shell) formed the basis of the validation of the current Cut 2 mine design, production schedule and financial analysis. The results of the Base Case pit optimisation are shown in Table 16-5. The Pit by Pit Analysis graph is shown in Figure 16-7.

Table 16-5: Pit optimisation results (base case pit by pit analysis).

Pit Shell	Pit value BWPm discounted			Pit tonnes (kt)		Recoverable carats (Kcts)	Life of mine (years)
	Best	Specified	Worst	Ore	Waste		
21	7,058	6,652	6,652	17,701	15,707	2,245	7.22
22	7,093	6,675	6,675	17,917	15,848	2,274	7.31
23	7,243	6,790	6,790	18,692	17,452	2,361	7.61
24	7,351	6,850	6,850	19,281	18,866	2,430	7.86
25	7,438	6,892	6,892	19,844	19,785	2,490	8.11
26	7,485	6,916	6,916	20,162	20,236	2,523	8.24
27	7,534	6,939	6,939	20,523	20,721	2,559	8.4
28	7,586	6,950	6,949	20,811	21,900	2,599	8.54
29	7,603	6,956	6,956	20,971	22,013	2,618	8.6
30	7,696	6,965	6,964	21,618	24,014	2,684	8.93
31	7,709	6,965	6,964	21,769	24,158	2,703	8.99
32	7,752	6,972	6,972	22,126	25,224	2,737	9.17
33	7,804	6,989	6,988	22,602	26,358	2,807	9.32
34	7,886	7,004	7,002	23,298	28,590	2,884	9.63
35	7,916	7,008	7,006	23,520	29,723	2,919	9.71
36	7,918	7,009	7,006	23,552	29,765	2,924	9.72
37	7,943	7,013	7,010	23,714	30,967	2,949	9.79
38	7,944	7,013	7,010	23,722	30,979	2,950	9.79
39	7,961	7,012	7,009	23,874	31,806	2,969	9.86
40	7,986	7,010	7,006	24,174	32,812	3,000	9.98
41	7,987	7,009	7,006	24,182	32,817	3,001	9.98
42	7,987	7,009	7,006	24,194	32,818	3,003	9.99
43	7,994	7,007	7,004	24,253	33,253	3,012	10.01
44	7,994	7,005	7,002	24,271	33,258	3,015	10.02
45	8,039	6,972	6,967	24,675	37,099	3,067	10.23
46	8,039	6,972	6,967	24,684	37,133	3,067	10.24
47	8,041	6,968	6,963	24,718	37,228	3,072	10.25
48	8,046	6,962	6,957	24,793	37,787	3,080	10.29
49	8,048	6,953	6,948	24,869	37,884	3,090	10.32
50	8,074	6,927	6,921	25,201	40,739	3,131	10.47

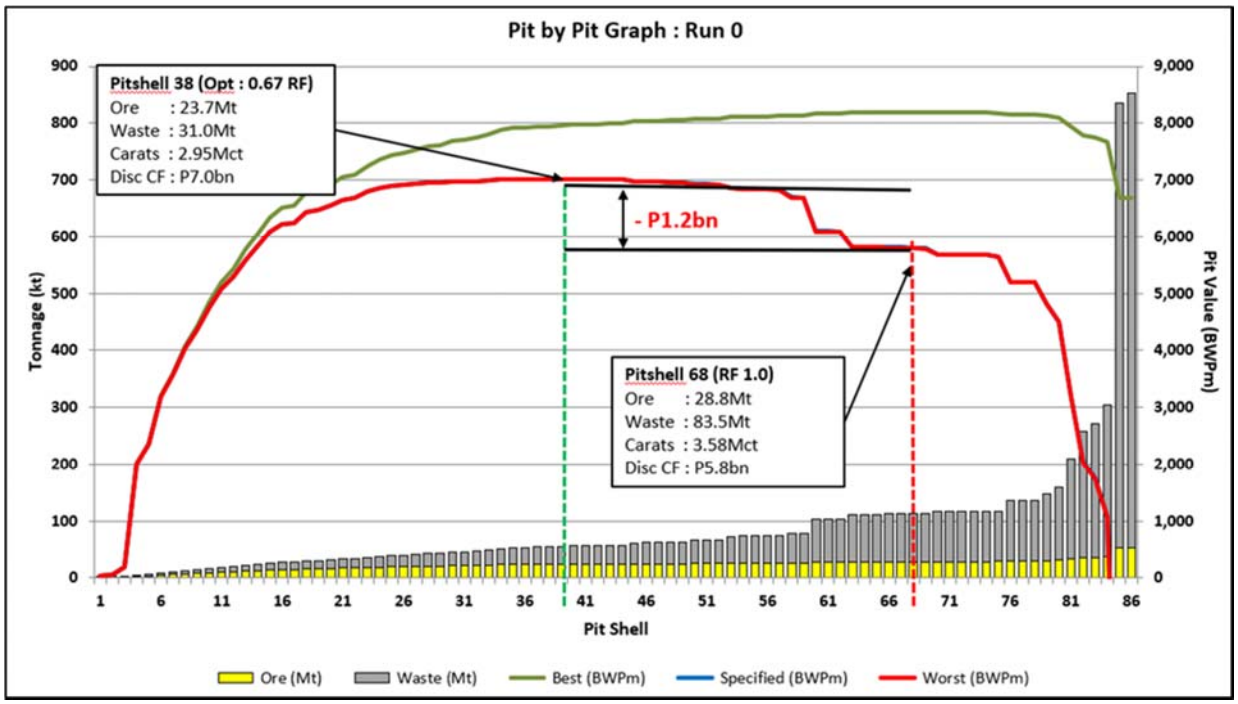

Figure 16-7: Pit by pit analysis graph for base case (Run 0).

The above pit by pit analysis graph (Figure 16-7) shows an optimal pit value at pit-shell 38. Pit shell 22 through to pit shell 59 demonstrate that the indicated discounted pit value changes within ±5 % of the peak pit value indicating the Karowe pit value appears relatively robust for a range of ultimate final shells (34 Mt – 78 Mt). A summary of the statistics associated with the selected pit (pit-shell 38) is listed in Table 16-6 below. No minimum mining width was applied during the sensitivity analysis.

Table 16-6: Pit-shell statistics for base case (run 0).

Parameter	Units	Optimal Pit-shell (38)	RF 0.51 Pit-shell (22)	RF 0.89 Pit-shell (59)	RF 1 Pit-shell (68)
Ore tonnes mined	Mt	23.7	17.9	26.2	28.8
Waste tonnes mined	Mt	31	15.8	51.8	83.5
Total tonnes mined	Mt	54.71	33.77	77.99	112.33
Life of mine	Yr's	9.8	7.3	11.1	13.1
Overall stripping ratio	Wt : Ot	1.31	0.88	1.98	2.9
Carats recovered	ct's	2,949,942	2,274,320	3,260,323	3,580,360
Average recoverable grade	cpht	12.43	12.69	12.45	12.43

16.3.22 Optimisation sensitivities

A number of pit size sensitivities were conducted to determine the robustness of the Karowe pit to changes in economic and technical conditions:

1. Run 0 – Base Case Parameters (as detailed in preceding section)
2. Run 1 - Diamond Price -25 %
3. Run 2 - Diamond Price +25 %
4. Run 3 - Mining Cost -25 %
5. Run 4 - Mining Cost +25 %
6. Run 5 - Processing Cost -25 %
7. Run 6 - Processing Cost +25 %
8. Run 7 - Recovery -20 %
9. Run 8 - Recovery -10 %
10. Run 9 - Slopes - 5 Degrees
11. Run 10 - Slopes + 5 Degrees
12. Run 11 - Discount Rate 5 %
13. Run 12 - Discount Rate 3 %
14. Run 13 - UG Cut-over (UG Cost of BWP 300 / tonne or US$ 29.27 / tonne)

A summary of the pit size sensitivity analysis completed in the pit optimisation is presented in Table 16-7 and is illustrated in Figure 16-8. All optimisation sensitivities for purposes of comparison did not include a minimum mining width.

Table 16-7: Optimisation sensitivity results.

Scenario	Description	Discounted cash flow (BWPm)	Ore (kt)	Waste (kt)	Recoverable grade (cpht)	Carats	Carats (Mct)	LOM (years)	Strip ratio
Run 0	Base Case	7,012.80	23,731	30,979	12.43	2,949,942	2.95	10	1.3
Run 1	Price -25 %	4,315.70	20,497	20,754	12.48	2,557,246	2.56	8	1
Run 2	Price +25 %	10,133.30	26,252	51,705	12.43	3,264,426	3.26	11	2
Run 3	Mining Cost -25 %	7,514.50	24,138	33,079	12.4	2,994,265	2.99	10	1.4
Run 4	Mining Cost +25 %	6,592.50	21,936	23,787	12.46	2,734,282	2.73	9	1.1
Run 5	Processing Cost -25 %	7,619.70	26,039	48,853	12.43	3,237,820	3.24	10	1.9
Run 6	Processing Cost +25 %	6,605.40	23,637	30,996	12.43	2,938,628	2.94	10	1.3
Run 7	Recovery -20 %	4,840.00	20,508	20,737	9.98	2,046,102	2.05	8	1
Run 8	Recovery -10 %	5,903.50	23,393	29,441	11.16	2,611,031	2.61	10	1.3
Run 9	Slopes - 5 Degrees	6,345.70	20,981	28,758	12.44	2,610,117	2.61	9	1.4
Run 10	Slopes + 5 Degrees	7,717.80	25,159	26,453	12.44	3,130,341	3.13	10	1.1
Run 11	Discount Rate 5 %	8,313.60	24,697	37,133	12.42	3,067,288	3.07	10	1.5
Run 12	Discount Rate 3 %	9,431.50	25,858	47,952	12.45	3,219,635	3.22	11	1.9
Run 13	UG Cut-over	6,941.10	22,117	22,302	12.34	2,729,409	2.73	9	1

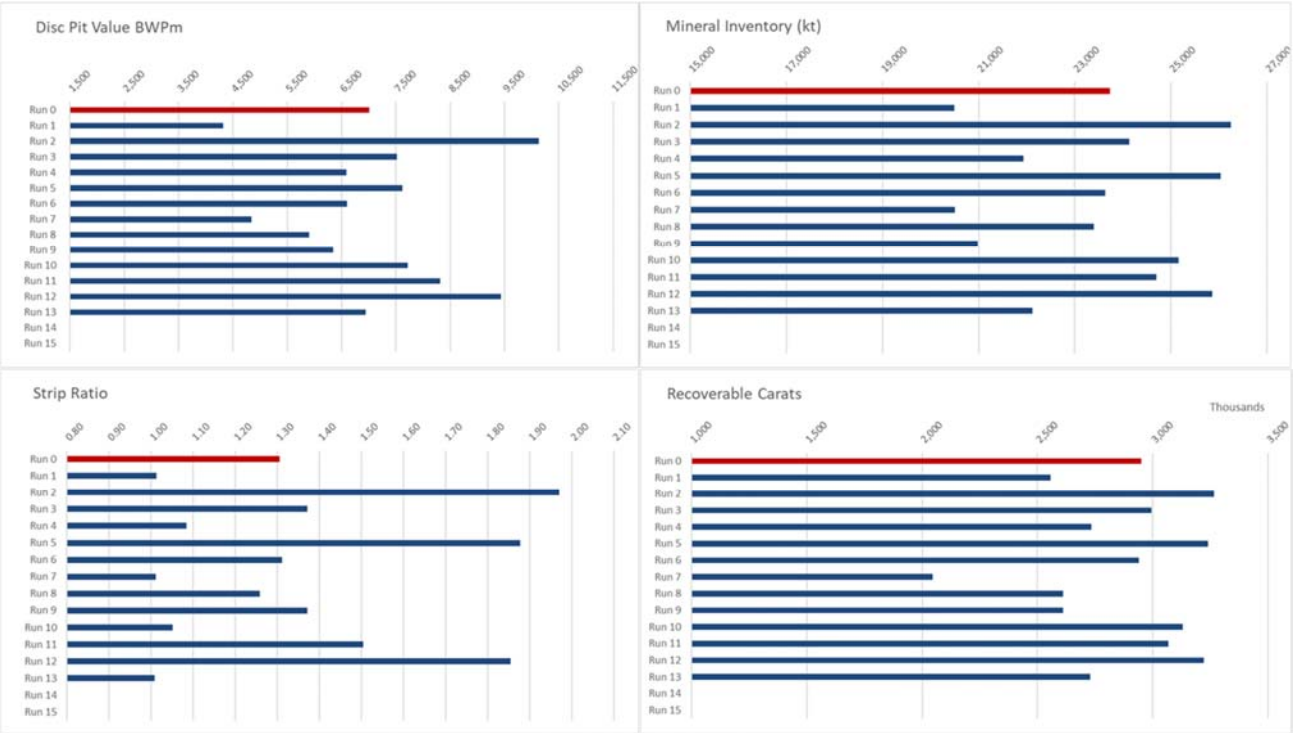

Figure 16-8: Optimisation sensitivities comparison.

16.3.23 Pit design parameters

Pit design parameters are in keeping with established mining practice and are described in the following sections. The pit design is based on Indicated Resources only.

16.3.24 Geotechnical considerations

Geotechnical slope design considerations were extracted from an SRK geotechnical report (Terbrugge and Mossop, 2017) for the Karowe Diamond Mine.

The update to the geotechnical slope recommendations focuses on the remaining rock-types to be mined in the current Cut 2 design, namely;

- Double benching the remaining fresh basalt;
- Sandstone; and
- Kimberlite Slope recommendations.

To complete the open pit optimisations, the SRK geotechnical recommendations for the other country rock strata from previous study work was used. The design parameters are detailed in Table 16-8 below.

Table 16-8: Mine design geotechnical considerations for all walls of each lithology.

Parameter	Calcrete	Weathered basalt	Fresh basalt	Sandstone	Mudstone	Kimberlite
Flitch height (m)	12	12	12	12	12	12
Number of flitches	1	1	2	1	1	1
Bench height (m)	12	12	24	12	12	12
Batter angle °	89	89	89	89	75	89
Berm width (m)	9.5	12	9	11	15	8
Inter-ramp angle (IRA)	51	44.5	69.7	48.6	34.9	57.1
Step-off (m)	-	-	0.5	-	-	-
Bench stack angle (BSA)	54.6	48.3	74.3	52	36.24	58.7
Geotechnical berm	-	-	-	-	-	-
Stack height (m)	12	12	24	12	12	12
Overall slope angle (OSA)	51	44.5	67.9	46.95	33.4	55.6

The overall slope angle (OSA) for each of the rock-types formed the basis of the design. No additional geotechnical berms have been recommended.

The pit slope recommendations outlined in the table above were reconfigured to provide for practical design and operational requirements based on equipment selection, grade control and blast design. However, overall slope angles have not been modified from the SRK slope recommendations.

16.3.25 Haul road design

The ramp width of an in-pit haul road should be at least 3.5 to 4 times the width of the selected haul truck. From the equipment on-site and the expected production volumes, the CAT 777 (or equivalent) is considered suitable for the expected mining volumes. Figure 16-9 illustrates the calculation of the required single and dual ramp widths used in the designs.

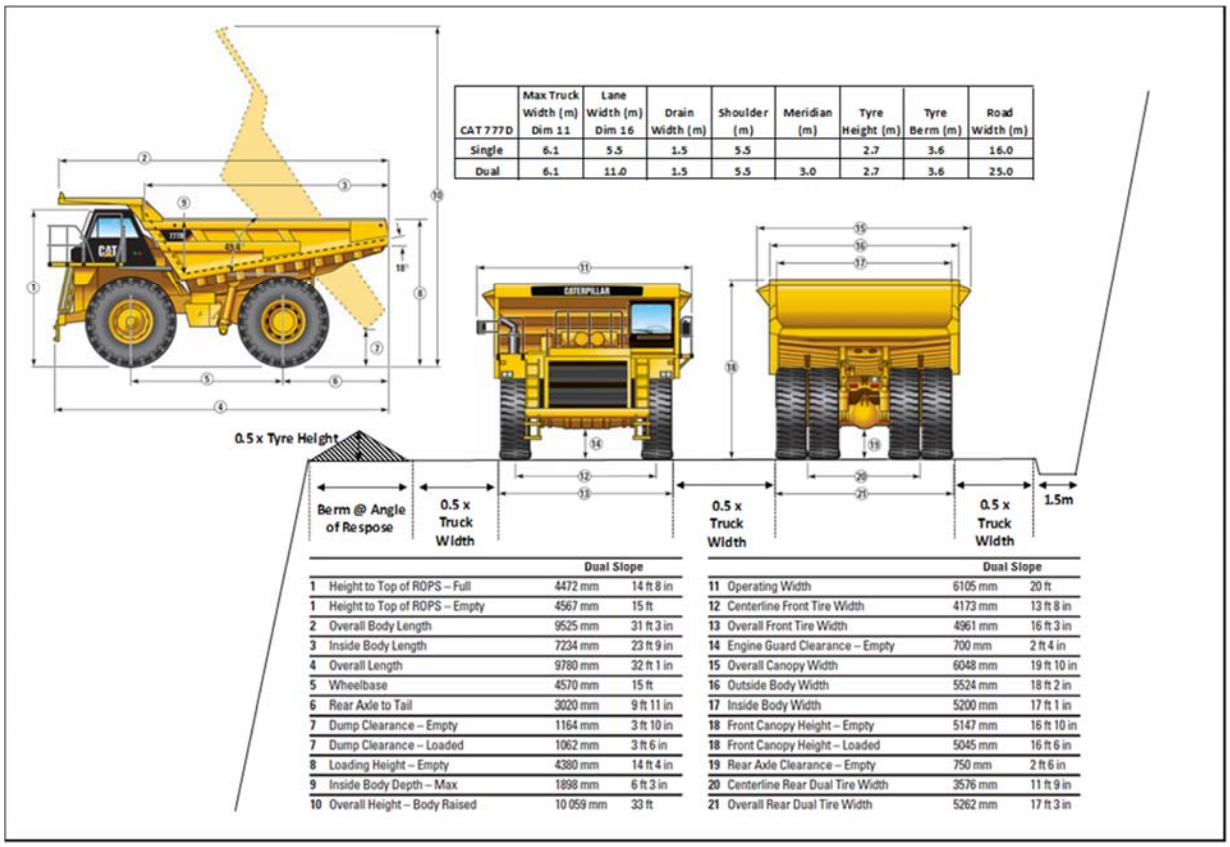

Figure 16-9: Haul road width design.

The pit exits for the various mining areas are positioned as a trade-off between proximity to waste dumps, ROM Pad, surface infrastructure, terrain at exit and minimal hauling distances. Pit design parameters are summarized in Table 16-9.

Table 16-9: Pit design parameters.

Pit design	Parameter	
Haul Road Design	Width – Dual lane	25 m
	– Single lane	16 m
	Gradient	10 %
	Minimum radius of turning circle	27 m
	Switchback Gradient	0 %
	Cross-fall Gradient	2 %
Working Widths	Minimum pit base width	50 m
	Minimum cutback width	~75 m

16.3.26 Optimal pit versus detailed design

In order to determine whether the final pit design is sufficiently similar in shape, size and position, to the original optimised pit, the potential ore and waste contents within the design are measured and compared

to the relevant open pit optimisation results (Table 16-10). As the open pit optimisation software employs a range of modifying factors in calculating the final output results, a similar process is followed when estimating the comparative figures from the pit design evaluation. A summary of this calculation and subsequent comparison is outlined below (Figure 16-10 to 16-13).

Table 16-10: Optimisation versus Cut 2 design variance.

	Ore (Mt)	Waste (Mt)	Total (Mt)	Strip ratio	Ore grade (cpht)	Recovered carats (Mct)
Optimised selected pit (Pit 38) Note 1	23.2	31.0	54.2	1.33	12.4	2.9
Detailed design Note 2	20.8	30.4	51.2	1.46	12.7	2.7
Variance % (optimisation vs design)	**10.4 %**	**1.7 %**	**5.4 %**	**-9.7 %**	**-2.7 %**	**8.0 %**

Note 1: Minimum mining width of ~70 m applied to base of ultimate pushback.

Note 2: Ore tonnes include in-situ VLG material.

Variances between the open pit optimal pit and the final pit design will invariably occur due to the application of design factors such as the ramp design parameters, as well as the detailed slope design. In the design, actual batter angles and berm widths are used, as opposed to the overall slope angle, (inclusive of a ramp system) as indicated in the pit optimisation results. Figure 16-10 through Figure 16-13 illustrate plan and cross-sectional comparisons between the Revenue Factor 1 pit-shell (red line), optimal pit-shell 38 (blue line) and current Cut 2 Ultimate pushback design (grey line).

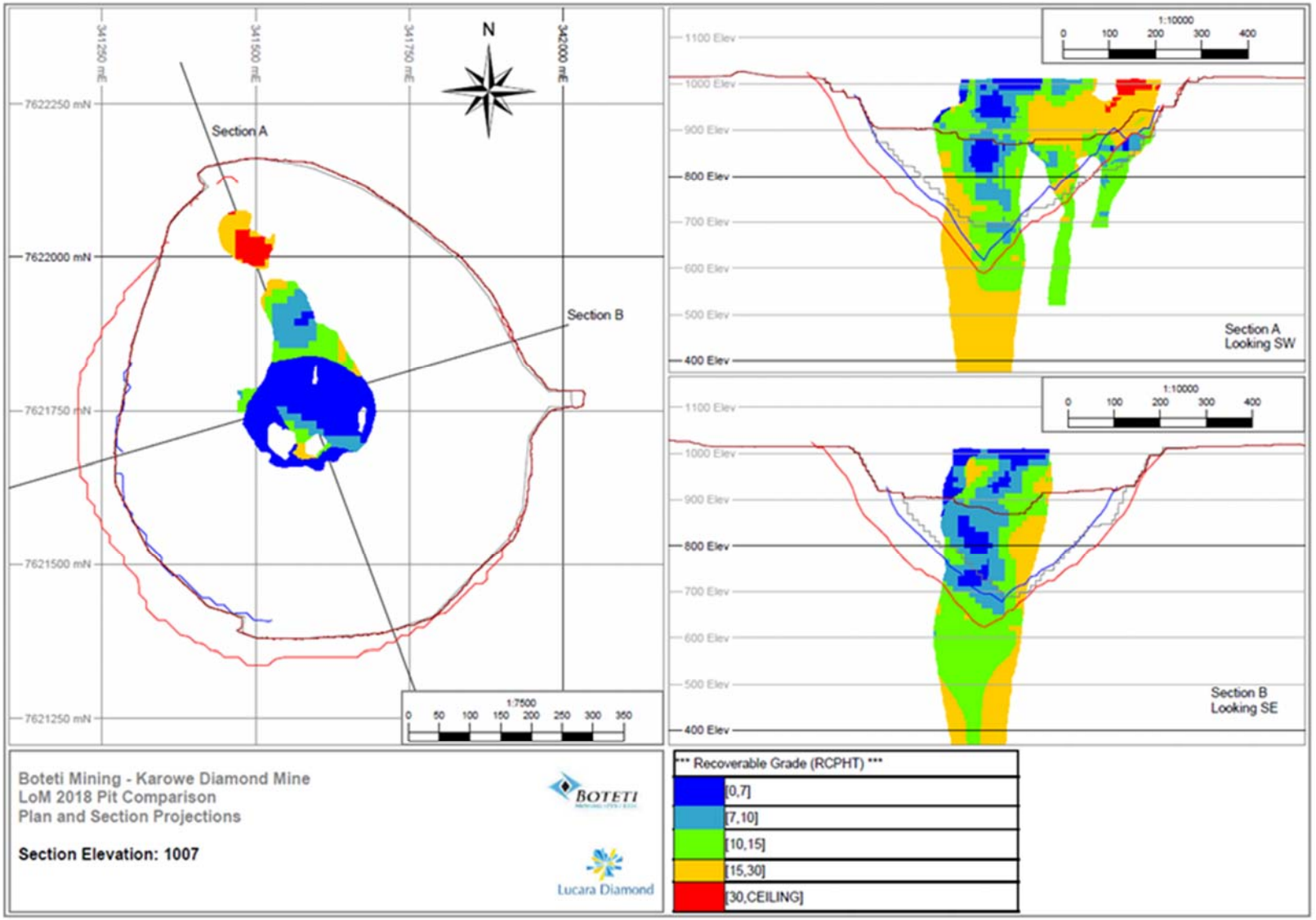

Figure 16-10: Optimal pit vs Cut 2 design comparison (horizontal section at 1007 masl).

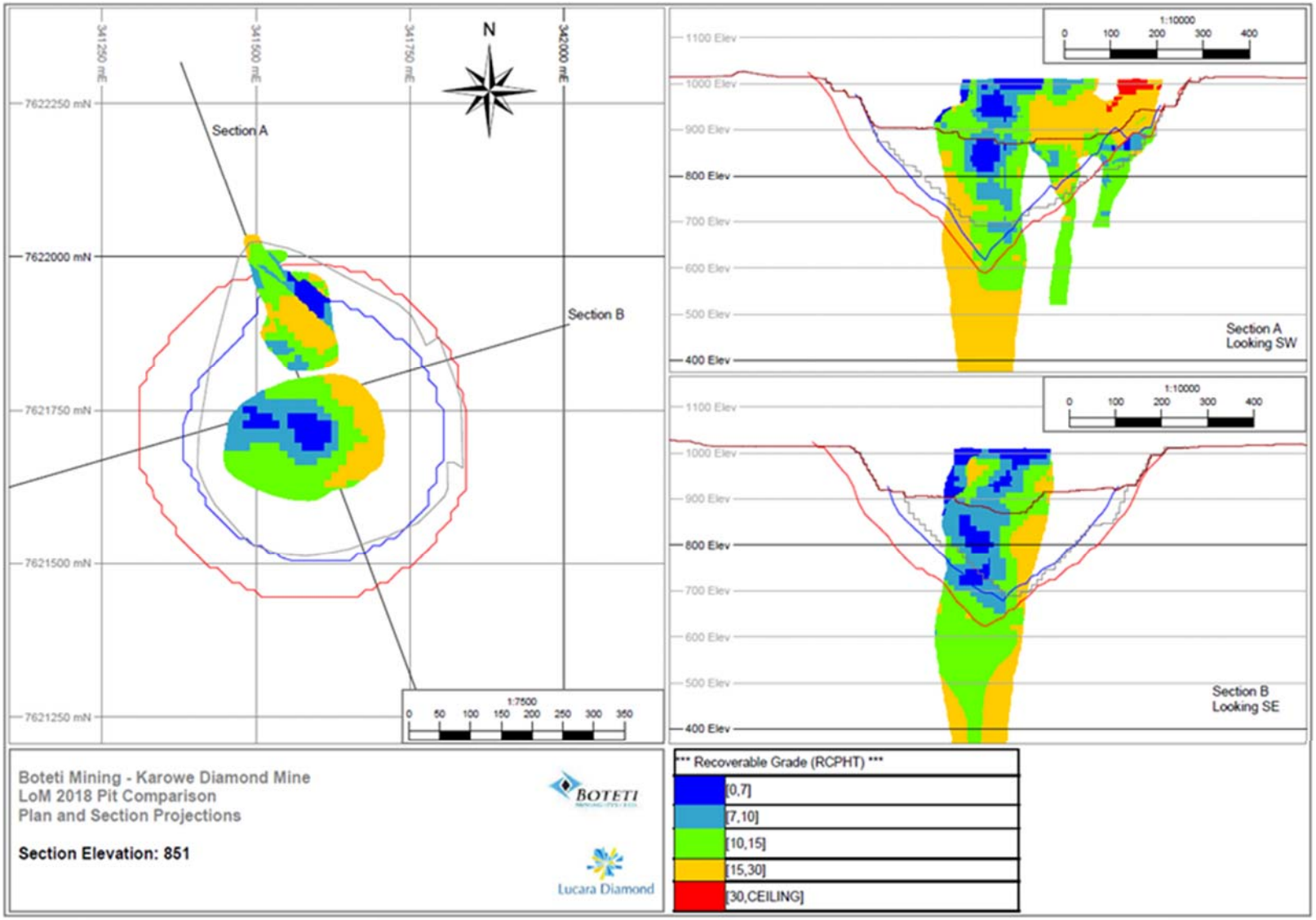

Figure 16-11: Optimal pit vs Cut 2 design comparison (horizontal section at 851 masl).

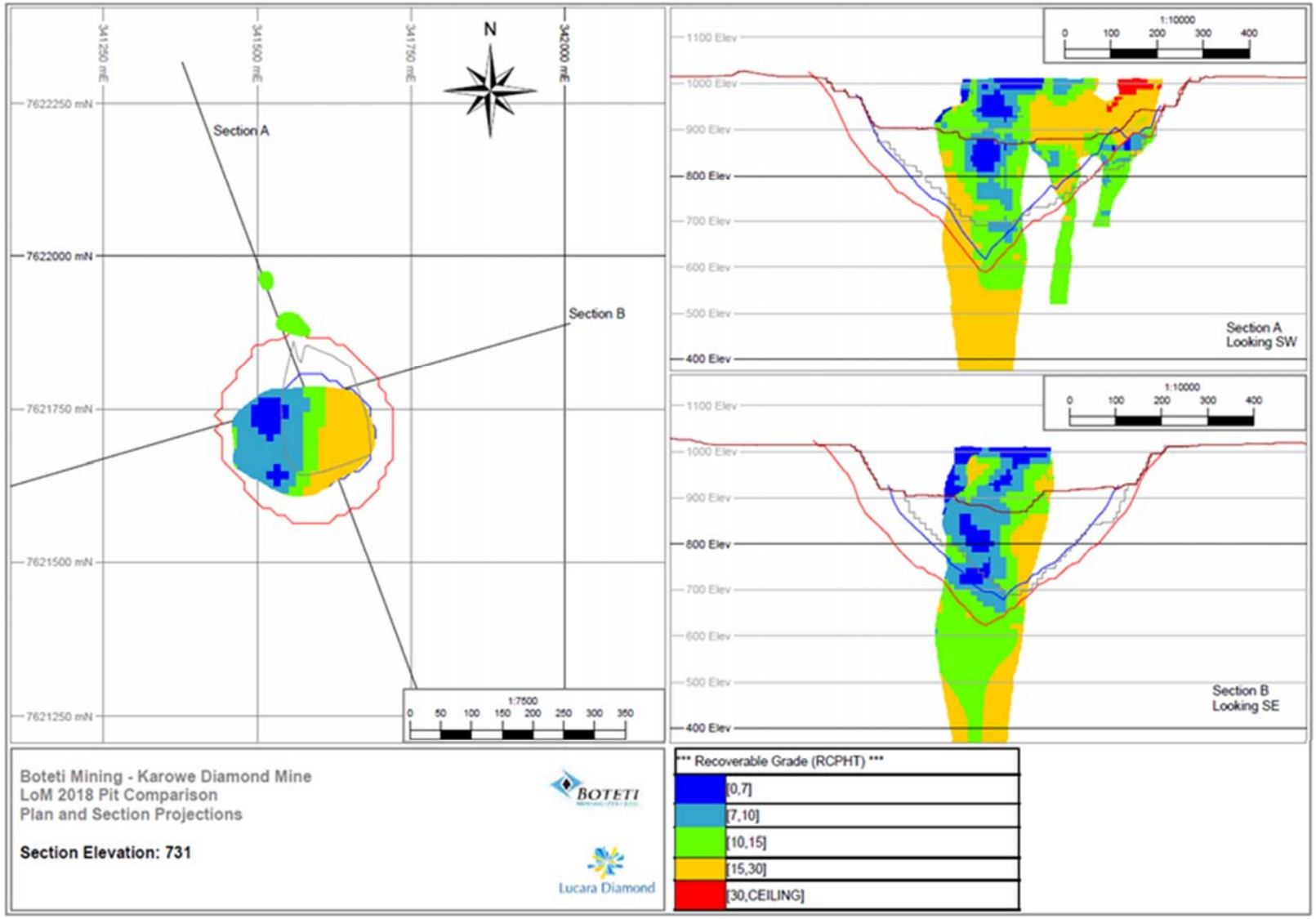

Figure 16-12: Optimal pit vs Cut 2 design comparison (horizontal section at 731 masl).

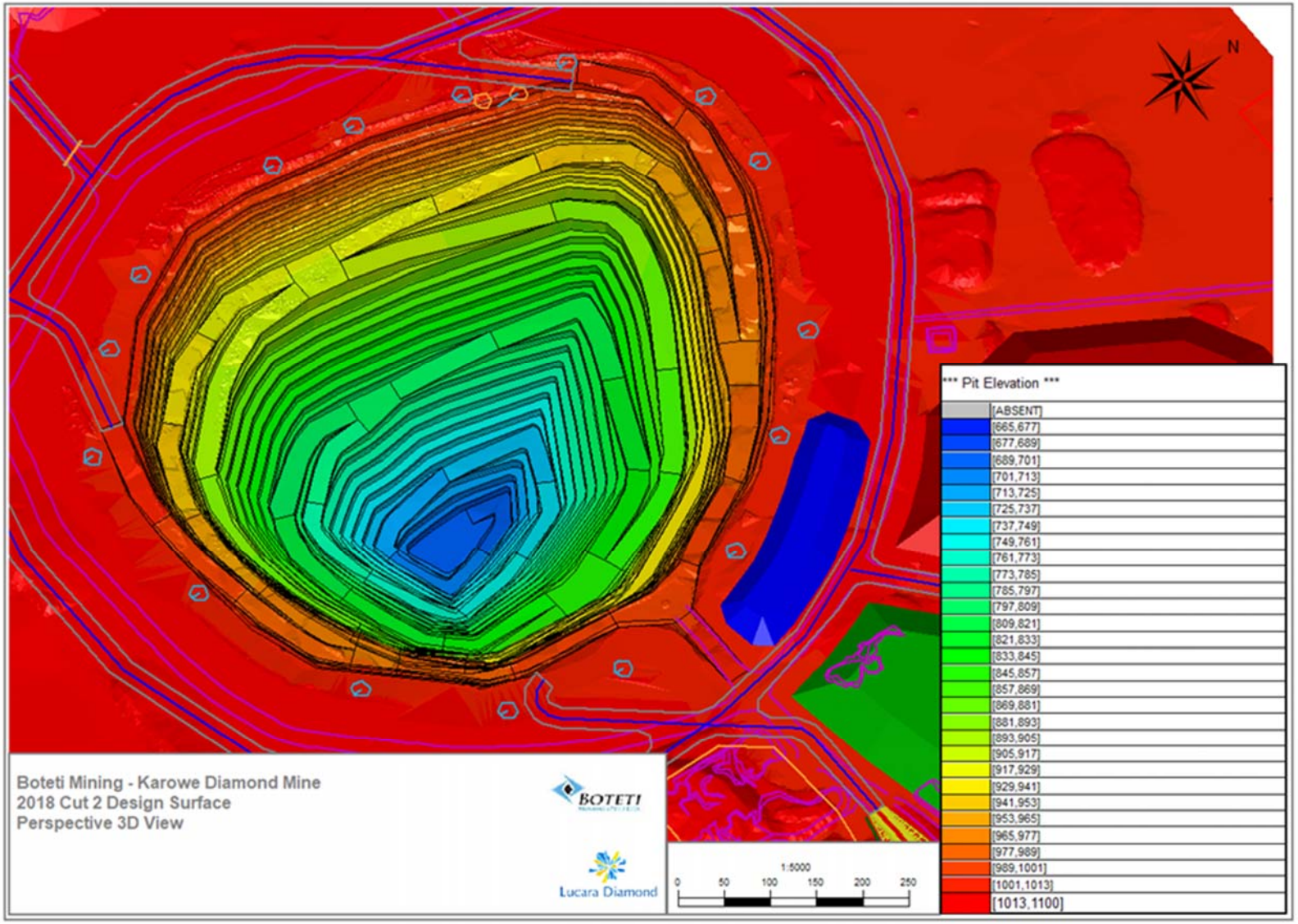

Figure 16-13: Perspective 3D view of the Cut 2 ultimate push-back design.

16.3.27 Grade control

Grade control drilling and sampling is not anticipated at the Karowe Diamond Mine. The ore material performance will be determined from the processing plant performance and interpolated into the geological resource model. Exploratory drilling in order to improve geological and grade confidence shall be completed by the on-mine technical departments and shall not form part of the mining activity.

Contamination of ore at the contact areas (hence dilution and ore loss) will be controlled and managed via the efficient mapping of the contact by the geological department, effective blast planning and execution of the blast layout as well as ore spotters assigned to the ore loading faces to visually examine and guide excavator operators when loading near the contact zones.

16.3.28 Pit dewatering and drainage

In-pit water management will consist of run-off control and sumps. The dewatering infrastructure and equipment is sized to handle ground water inflows and precipitation.

The in-pit dewatering plan is based on diverting as much surface water as possible away from the open pits, then collecting the water that does report to the open pits, using ditches and sumps before pumping it to the Mine Water Pond. There will be intermediate sumps on the pits walls as well as on the surface between the pit and the Mine Water Pond.

As the LOM pit will be operating at depths greater than 300 m below crest, specialist high lift pumps will be required. Pontoon mounted pumps will be used to draw from sumps. This will ensure the pumps are not submerged as sump water levels rise rapidly in response to a rainfall event. Pumping infrastructure will advance with the active mining as it advances deeper.

The key operational requirements will be to:

Minimise water flows into the pit using perimeter bunds, drains and fill, where practicable:

- provide pit pumping capacity for foreseeable extreme events;
- maintain pit wall drainage;
- provide permanent and temporary sumps capable of handling the expected water inflows; and
- install settling ponds for the removal of solids prior to discharge off-site.

The ex-pit dewatering plan currently utilises 16 deep dewatering boreholes located on the periphery of Cut 2 design to depress groundwater and hence reduce groundwater inflows into the pit. The current dewatering bores are considered of strategic importance for the make-up water for the processing plant.

16.3.29 Human settlement considerations

The Karowe Diamond Mine has no significant human settlements in the vicinity of the mining and processing activities.

16.3.30 Waste rock dumps

The waste rock dumps associated with mining operations will continue to be constructed to meet the requirements of the Botswana Mining Regulations and international best practices. They will initially to be constructed with the natural rill angle of approximately 37°, which is the angle of repose of the dumped material. This is then to be contoured progressively to an overall slope angle of 18.5° (1:3) to allow for slope stability and re-vegetation. The waste dump will be progressed by tipping from a higher level against a windrow and progressively pushing the waste out with a dozer.

Approximately 21 million m^3 will be required to store 31 Mt of waste country rock. The current waste rock dump has sufficient capacity considering a dump expansion to the west and has been designed to accept the remaining waste in the Cut 2 design (Figure 16-14). Should additional storage be required an expansion northward of the current north west dump may be considered.

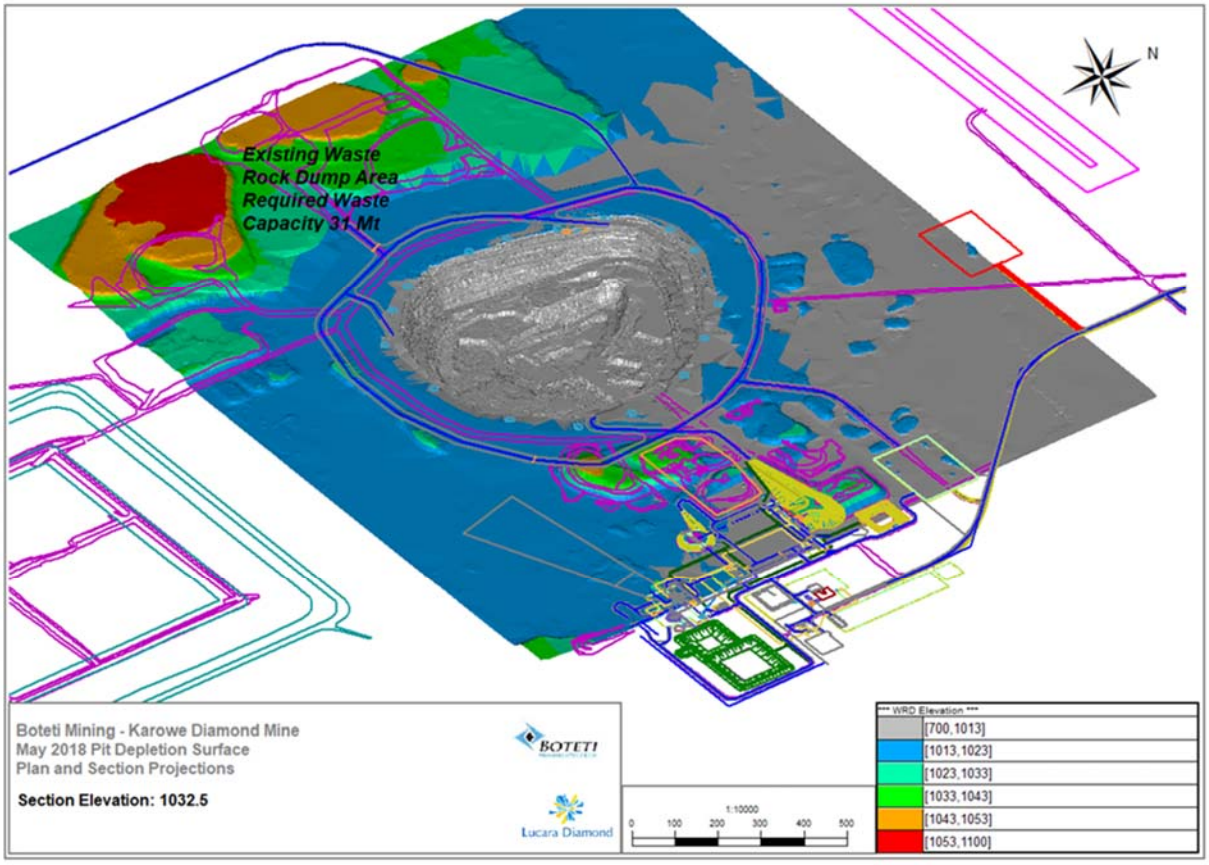

Figure 16-14: Waste rock storage.

16.3.31 Stockpiles

Karowe Diamond Mine currently makes use of stockpiles as part of the processing plant feed management programme. The surface stockpiles can be aggregated into 5 main categories, namely:

1. LOM (<7 cpht)
2. Contact Zone Material
3. Low Grade (7 – 10 cpht)
4. Medium Grade (10 – 15 cpht)
5. High Grade Grade (+15 cpht)

The opening balances of the surface stockpiles are detailed in Table 16-11.

Table 16-11: Stockpile opening balances as of May 2018.

Stockpile	Stockpile opening balance (Mt)	Stockpile opening balance (ct)	Stockpile grade (cpht)	Peak balance 2024 (Mt)
LOM (<7 cpht)	2,487,259	97,388	3.9	3,464,518
Contact Zone Material	256,691	21,779	8.5	256,691
Low Grade (7 – 10 cpht)	853,932	72,926	8.5	3,923,080
Medium Grade (10 – 15 cpht)	900,547	97,952	10.9	1,375,580
High Grade (+15 cpht)	91,856	16,799	18.3	91,856
Total surface stockpiles	**4,590,286**	**306,844**	**6.7**	**9,111,726**

It is planned to use the same stockpiling strategy for the remaining Mineral Reserve. LOM stockpile will only be processed at the end of life of the open pit mining operation. Current stockpile balances are planned to double by 2024 as a result of accelerate ore mining from 2021 to 2024 when it is planned for drill and blast, load and haul activities to stop by the end of 2024 calendar year (Figure 16-15). Accelerated ore mining has been planned to allow for potential underground ore production to commence with long hole blast-hole retreat mining of stopes in the kimberlite pit walls at the base of the pit. Reclamation of the stockpiles will be used to dove-tail the ramp-up ore production from the potential underground and the plant feed requirements whilst maintaining the plant throughput to 2.6 Mtpa (Figure 16-16).

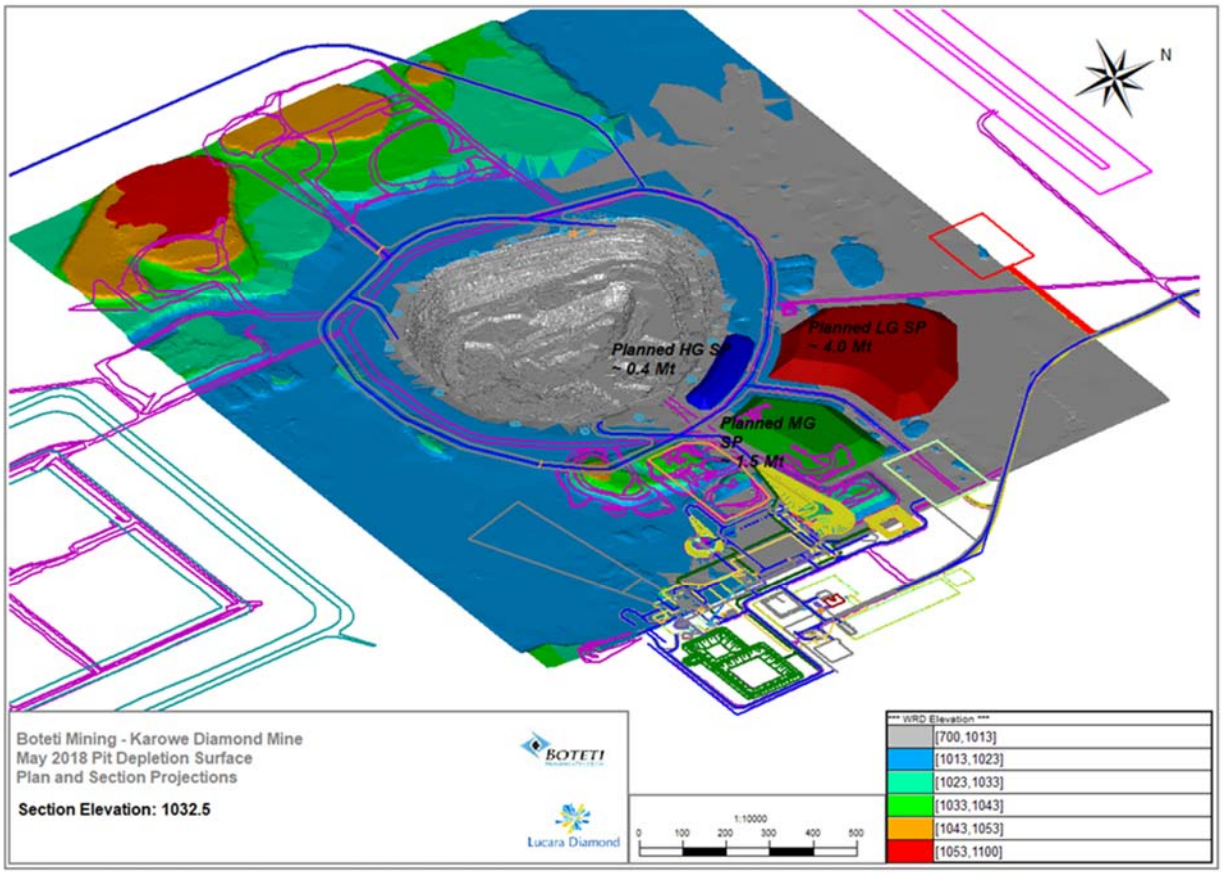

Figure 16-15: Planned stockpiling area (LG, MG, and HG).

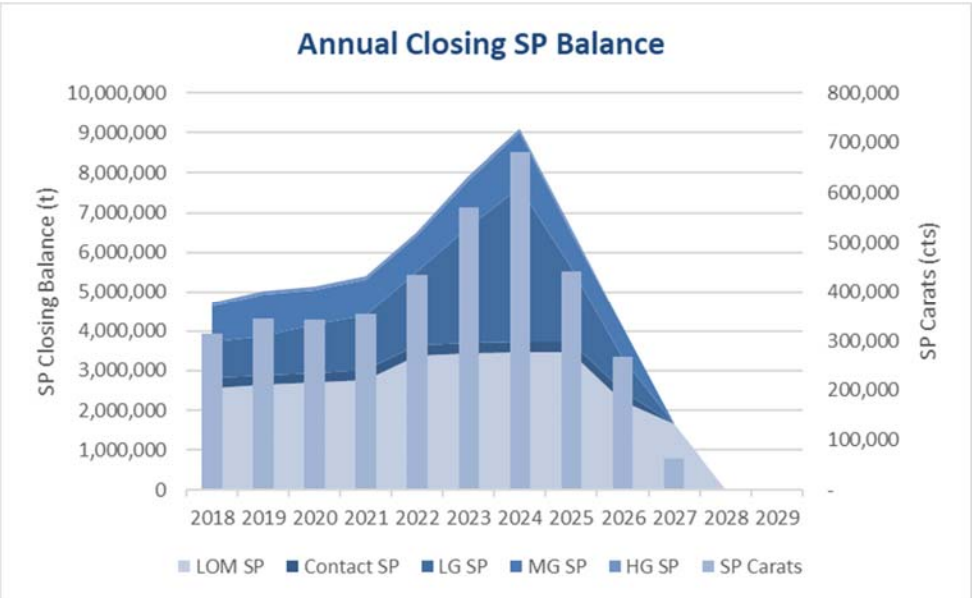

Figure 16-16: Surface loose and stockpile closing balances.

16.3.32 Mining equipment

Karowe Diamond Mine uses contractor mining services for drill and blast, load and haul, re-handle and pit support services.

16.3.32.1 Load and haul equipment

The mining fleet consists of a mix of smaller classed equipment used for ore and re-handling and larger equipment used primarily for waste stripping. The larger classed waste fleet consists of 100 to 120 t class hydraulic excavators with bucket capacity of 4-5 m^3 in backhoe configuration, and 90 to 100 t payload off highway rigid frame dump trucks. The smaller classed ore fleet consists of 70 to 90 t class hydraulic excavators with bucket capacity of 4 to 5 m^3 in backhoe configuration, and 45 t payload off highway articulated dump trucks.

The primary mining fleet of trucks and excavators will be supported by standard open-cut drilling and auxiliary equipment.

Waste material will be hauled to the allocated waste rock dump positions to the west of the pit. Ore will be hauled and dumped at the assigned stockpiling area or direct fed into the primary ore crusher located to the east of the pit.

16.3.32.2 Drill and blast equipment

Rock fragmentation will be accomplished through drilling and blasting. All fresh competent kimberlite and waste material will be drilled and blasted.

The drill and blast activity will be supported with a mixed fleet of Atlas Copco 660, Ingersol Rand DM30 and Sandvik D25KS drill rigs capable of drilling 102 – 152 mm vertical and inclined holes. The rigs will be supported by a stemming FEL and explosive delivery vehicle and several special purpose LDV's carrying personnel and explosive accessories.

The pit configuration bench height and waste material type anticipated at the project suit drill rigs capable of drilling drill holes with a diameter of 154 mm. Drill burden, spacing and sub-drill design will be functions of the varying material types of the deposit.

An emulsion-based product with water resistant characteristics and a higher velocity of detonation is recommended to achieve a better fragmentation.

The blast pattern is dictated by the powder factor required to ensure appropriate fragmentation and heave. The selection of the powder factor is based on the Unconfined Compressive Strength (UCS) measurement results.

As part of the geotechnical optimisation of the pit, pre-split blasting will be required for the complete final wall position. The pre-split cost has been included in the operating cost. The pre-split holes will be drilled at a spacing of 1.5 m (12 m deep) with a hole diameter of 127 mm and a 15 m buffer trim shot will blasted

in conjunction with the pre-split final wall. The pre-split blasting will achieve two goals, reduction of ground vibration and protection of the high wall condition.

16.3.32.3 Pit support equipment

Pit support equipment for the Karowe Diamond Mine operation will consist of dozers, graders, fuel bowsers, water bowsers, hydraulic hammers, and wheel loaders. The function of this equipment will be to support the primary mining equipment through maintenance of pit floor and haul road conditions, provide clean-up around the excavators to prevent excessive tire damage, secondary breakage of oversize rocks and to water-down road surfaces to supress dust.

The majority of the plant feed material is planned to be loaded directly from the pit into the primary crusher during the period 2018 to 2021. During the period 2022 to 2024 accelerated ore mining in the pit will necessitate stockpiling of approximately 33 % of the ROM ore tonnage in preparation for the potential underground mining operations. It has been assumed that a small buffer stockpile located at the ROM Pad in close proximity to the ore feed bin will be maintained at 10 % of the total monthly feed tonnage.

Re-handle equipment for the operation will consist of a combination of FEL and 80 t class excavators loading and 40 to 45 t class ADT haul trucks when re-handling from the stockpiles to the ROM feed bin.

16.3.32.4 Ancillary equipment

Ancillary equipment for the operation will consist of service trucks, tyre handlers, mobile crane, water pumps, lighting plants, TLB, LDV's and wheel loaders. The function of this equipment will be to support the pit equipment and maintenance workshops.

16.3.32.5 Mining equipment summary

Mine Equipment is summarised in Table 16-12 below. The mining fleet is expected to ramp-down as waste stripping diminishes over the LOM.

Table 16-12: Mining equipment summary.

Area	2018 (17.5 Mtpa)	2020 (10 Mtpa)	2022 (6.5 Mtpa)
Pit load and haul (primary fleet)			
100 - 120 t class excavator	3	2	0
90 - 100 t payload haul trucks	11	11	0
80 - 90 t class excavator	1	1	3
40 - 45 t payload haul trucks	10	15	27
Pit load and haul (pit support fleet)			
40 - 50 t class track dozer	3	3	2
100 t class track dozer	2	2	2
Hydraulic hammer - excavator 20 t	2	2	2
Water bowser	1	1	1
30 t class wheel dozer	1	1	1
Motor grader	1	1	1
FEL	1	1	1
Drilling fleet			
D25KS - drill rig	3	2	2
ATLAS COPCO 660 - drill rig	4	3	2
DM30 - drill rig	4	3	2
Stockpile ore to crusher			
40 t class excavator	2	1	1
50 t payload truck	3	3	3
FEL	1	1	1
45 t class track dozer	1	1	1
Ancillary fleet			
40 t water bowser	1	1	1
Motor grader	1	1	1
Tyre handler (workshop)	1	1	1
Total equipment	**57**	**57**	**55**

16.4 Life of mine and production rates

16.4.1 Mining schedule

The open pit LOM mine production schedule is illustrated in Figure 16-17 below. A higher stripping ratio is required in 2018 to prevent a bottle-neck in ore production in subsequent years. Mining production requirements in 2019 and 2020 will reduce from ~17 Mtpa to 10 Mtpa. From 2021, the mining production requirements will be approximately 6.5 Mtpa until the end of life of the open pit mining operation. Ore production has been accelerated from 2021 until end of life in 2024 in order to ensure that no open mining activities are required at the base of the pit when potential long-hole blasting of the pit-wall kimberlite is planned to start ramping-up in 2025. The additional ore production beyond the 2.6 Mtpa processing requirements will be stockpiled for re-handle into the processing plant once mining activities have ceased

in the open pit and will augment potential underground ore production during the ramp-up phase of the potential underground mine (Figure 16-17).

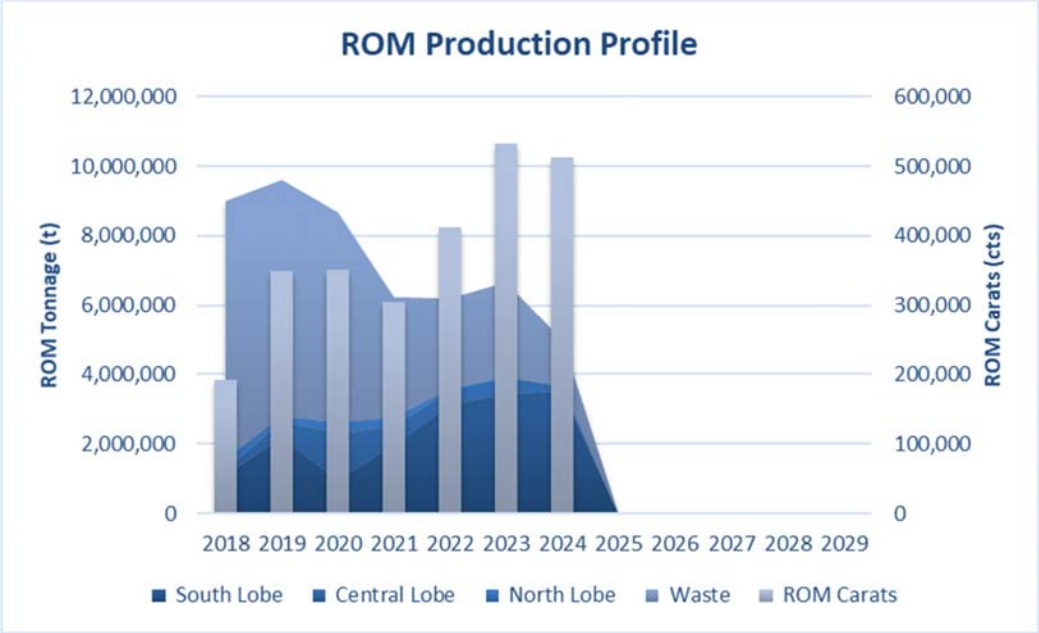

Figure 16-17: Open pit LOM production profile. Note: production represented for 2018 is for June 2018 to December 2018 and does not include actuals for the period January 2018 to May 2018.

16.4.2 Plant feed schedule

The plant feed schedule is based on the design throughput of the processing plant (2.5 Mtpa) considering the maintenance and planned shut-down schedules as planned by Karowe Diamond Mine. Figure 16-18 below shows the processing plant feed tonnage schedule for the open pit mining operation with re-handle of surface loose stockpiles at the end of the open pit mining production.

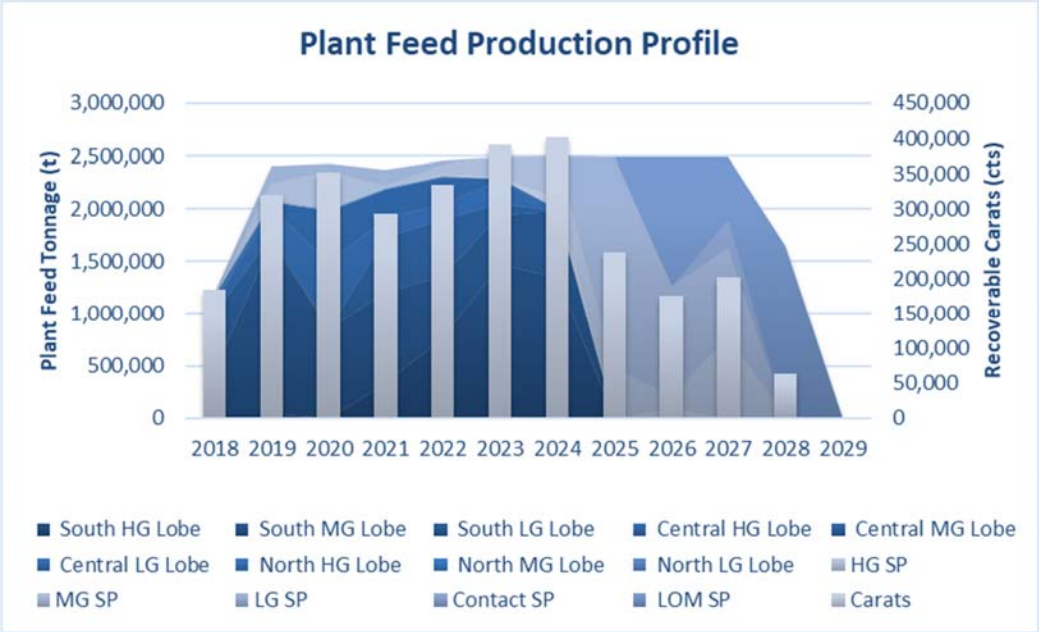

Figure 16-18: Plant feed schedule. Note: plant feed represented in graph for 2018 is for June 2018 to December 2018 and does not include actuals for the period January 2018 to May 2018.

17. Recovery methods

This section is contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. The information documented herein was extracted and summarized from Oberholzer et al. (2017).

17.1 Mineral processing plant

The Phase 2 upgrade is described in Section 13 and the resulting Process Diagram is shown in Figure 17-1.

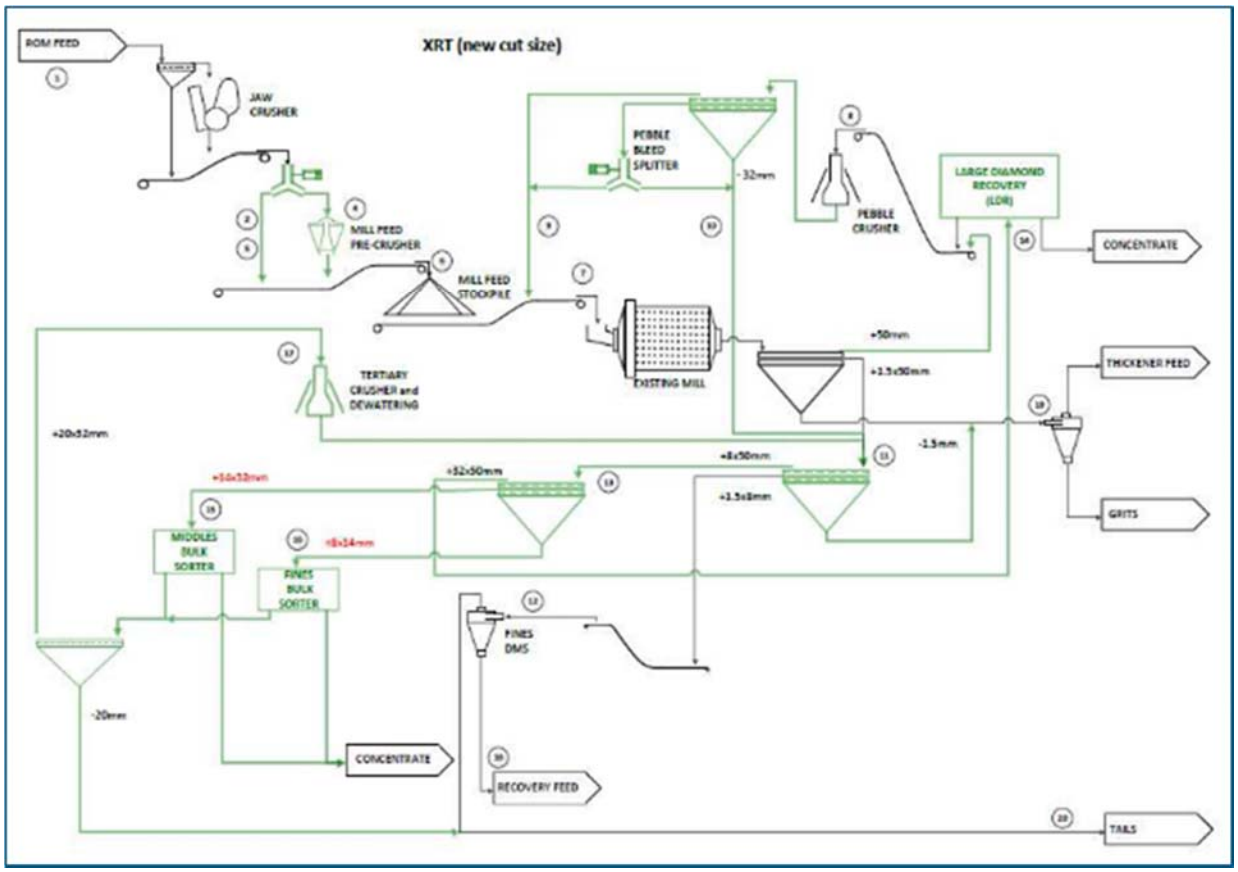

Figure 17-1: Phase 2 process flowsheet.

The Phase 2 upgrades were intended to:

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

Phase 3 process upgrades have been completed and have been in operation since Q3 2017. These upgrades include an XRT circuit treating 50 to 125 mm 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 to 8 mm 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 resources.

18. Project infrastructure

This section is contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. The information documented herein was extracted and summarized from Oberholzer et al. (2017).

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

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 Workshops

A workshop on surface currently services the open pit mining fleet.

18.1.3 Water handling

The existing volume of water from the boreholes provides sufficient water for the process plant and dust suppression.

18.1.4 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 15 MVA 33/11 kV stepdown transformers are installed at Karowe. The primary reticulation on the mine site is at 11 kV.

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

18.2 Tailings storage facility

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 1041 masl.

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.

19. Market studies and contracts

This section is contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. The information documented herein was extracted and summarized from Oberholzer et al. (2017), updated where relevant to March 2018.

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 (+10.8 ct 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 infrequent 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 (EST) 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$250,000 to approximately US$1 million. 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.

Historically, Lucara has sold diamonds through both regular stone tenders (RST's) and exceptional stone tenders (EST's). Diamonds that qualify for EST's are rare, selected on a range of criteria including weight, quality, color, and, often achieve sales prices in excess of USD$ 1 million per diamond. On average, Lucara has held between 4 and 5 RST's and 1 to 2 EST's per annum.

Lucara continues to adjust its sales strategy to maximize client participation and achieve best possible revenue. As a result, Lucara has decided to conduct an exceptional stone tender (EST) during the regular tender scheduled for June 2018 and thereafter, will move to a blended tender process, whereby a greater number of exceptional stones will be sold as part of RST's. This will decrease the inventory time for large, high value diamonds and will generate a smoother, more predictable revenue profile that better supports price guidance on a per sale basis.

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 that came online through 2016 and 2017 (Renard, Gahcho Kue, Liqhobong) achieved market prices for new production that 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 community impact

This section is contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. The information documented herein has been summarized from Oberholzer et al. (2017). There have been no significant changes with respect to aspects covered by this section.

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 2007 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 the Karowe Diamond Mine (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, 2018). 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, 2018). 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, 2018).

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 and 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 per annum with annual abstraction over the past 5 years around 2.5 million cubic meters (Geoflux, 2018). 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, 2018). 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, 2018).

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, 2018). 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$19.4 million (Geoflux, 2018). 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.

21. Capital and operating cost

This section was contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. The section is based on actual 2017/18 costs and budget inputs from Karowe Mine.

Operating costs are presented in Section 16.3.8 and are based on the current Whittle® optimisation and mining schedule, which in turn defines the Mineral Reserve for the open pit. The mine schedule is designed to deliver approximately 2.5 Mt to the mill on an annual basis. The schedule involves termination of open pit mining in 2024, with ongoing processing of surface stockpiles through to the end of 2028. Cessation of open pit mining dovetails with a potential underground mining operation currently at the Feasibility level of study.

Capital costs for the mine are included in the model at an estimated USD 9.3 million per year for the remainder of the Cut 2 design open pit, with mine closure expenses of USD 19.4 million. The mine is currently undertaking a feasibility study for a potential underground mining operation after the completion of a preliminary economic assessment (Oberholzer et al., 2017). Pending the results of the current feasibility study additional capital expenditures may be forthcoming.

22. Economic analysis

This section was contributed by Lucara Diamond Corp. under the oversight of Dr John Armstrong. An updated financial model has been produced incorporating the following inputs:

- The current Mineral Reserve estimate based on the optimised Whittle® Shell with mining to 2024 and stockpile feed to 2028;
- The 2018 Lucara Rough Diamond Price Book has been used as a basis for diamond value;
- An assumed increase in diamond value (real) of 2.5 % for a period of four years;
- An annual increase in mining costs to allow for longer haul distances as the pit deepens
- Taxes and royalties as they currently apply (Section 14.1.2);
- Lucara Botswana administrative cost of USD 11 Million per annum;
- Sales and marketing costs of 1.9 % of sales.

The cash flow analysis is presented in Table 22-1. The net present value of the open pit portion of the project at an 8 % discount rate is USD 480.8 million. On a pre-tax basis with an 8 % discount rate the NPV is USD 786.1 million.

Table 22-1: Summary of financial model (Page 1 of 4).

FA Rates:	
USD/PULA FX RATE	10.25

Net Present Value	NPV (USDm)
0%	$658,022
5%	$536,732
8%	$480,754
10%	$448,790
15%	$383,526

			Total (undiscounted)	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
CASH FLOW														
Revenue (USDm)														
North Lobe			31,970	13,008	4,922	4,464	3,798	1,075	-	-	-	-	-	4,703
Centre Lobe			192,128	7,507	20,454	75,353	24,524	25,072	17,873	3,597	14,309	-	-	3,438
South Lobe			1,522,323	75,045	179,453	103,869	165,966	211,377	273,653	311,244	63,704	55,030	68,456	14,525
Working Stockpile			152,823	-	-	-	-	-	-	-	109,581	43,243	-	-
LOM			112,446								-	31,193	46,903	34,349
Total Revenue			**2,011,690**	**95,560**	**204,829**	**183,686**	**194,289**	**237,524**	**291,527**	**314,841**	**187,593**	**129,466**	**115,359**	**57,016**
Costs (USDm)														
Ore mined			76,057	4,877	8,836	8,270	9,021	12,944	15,556	16,553	-	-	-	-
Waste Mined			100,848	22,132	21,103	19,638	11,944	9,446	10,807	5,779	-	-	-	-
Mining Contractors Fixed Manangment Fee			26,482	2,037	4,074	4,074	4,074	4,074	4,074	4,074	-	-	-	-
Ore Incremental Mining Cost			6,660	504	894	820	850	1,150	1,266	1,175	-	-	-	-
Processing			269,791	15,509	26,441	26,526	26,441	26,441	26,441	26,526	26,441	26,441	26,441	16,142
Site administration			88,058	5,527	11,055	11,055	11,055	11,055	11,055	11,055	4,051	4,051	4,051	4,051
Total site operating costs (USD M)			**567,897**	**50,587**	**72,403**	**70,383**	**63,385**	**65,110**	**69,199**	**65,161**	**30,492**	**30,492**	**30,492**	**20,193**
Site Costs (ex Mining)	**$ / t**		**15.39**	**16.15**	**17.06**	**17.01**	**17.04**	**17.16**	**17.20**	**17.15**	**12.25**	**12.25**	**12.25**	**13.29**
Site Costs (in Mining)	**$ / t**		**22.35**	**34.64**	**29.08**	**28.18**	**25.46**	**26.15**	**27.79**	**26.09**	**12.25**	**12.25**	**12.25**	**13.29**
Karowe gross margin			**1,443,793**	**44,973**	**132,426**	**113,303**	**130,904**	**172,414**	**222,328**	**249,679**	**157,101**	**98,974**	**84,867**	**36,823**
	Royalty													
Marketing Fees	1.9%		38,222	1,816	3,892	3,490	3,691	4,513	5,539	5,982	3,564	2,460	2,192	1,083
Royalties	10%		201,169	9,556	20,483	18,369	19,429	23,752	29,153	31,484	18,759	12,947	11,536	5,702
Karowe cash operating margin			**1,204,402**	**33,602**	**108,051**	**91,445**	**107,783**	**144,149**	**187,636**	**212,213**	**134,778**	**83,568**	**71,139**	**30,038**
Boteti Admin costs	0.0%	12	35,205	1,676	3,353	3,353	3,353	3,353	3,353	3,353	3,353	3,353	3,353	3,353
Net income / (loss) before tax			**1,169,197**	**31,925**	**104,698**	**88,092**	**104,430**	**140,796**	**184,283**	**208,860**	**131,425**	**80,215**	**67,786**	**26,685**
Income Taxes			441,830	5,979	35,914	27,461	37,308	56,279	78,619	92,326	57,208	30,081	19,996	658
Withholding tax			-	-	-	-	-	-	-	-	-	-	-	-
Previous period tax payments														
Finance income / cost														
Net profit			**727,367**	**25,947**	**68,784**	**60,630**	**67,122**	**84,517**	**105,664**	**116,534**	**74,216**	**50,135**	**47,790**	**26,028**
Capital Expenditures	**Annual Increase**	**# Yrs (escalation)**												
Sustaining capital	0.0%	0	99,750	4,750	9,500	9,500	9,500	9,500	9,500	9,500	9,500	9,500	9,500	9,500
Rehabiliation			19,195	-	-	-	-	-	-	-		-	**5,000**	**14,195**
Total capital expenditures			**118,945**	**4,750**	**9,500**	**9,500**	**9,500**	**9,500**	**9,500**	**9,500**	**9,500**	**9,500**	**14,500**	**23,695**
Karowe Free Cash Flow			**608,422**	**21,197**	**59,284**	**51,130**	**57,622**	**75,017**	**96,164**	**107,034**	**64,716**	**40,635**	**33,290**	**2,332**
Working capital movements			**-**											
Total cash flows (USD)	**8%**		**608,422**	**21,197**	**59,284**	**51,130**	**57,622**	**75,017**	**96,164**	**107,034**	**64,716**	**40,635**	**33,290**	**2,332**
Beginning cash (USD,000)				**49,600**	**70,797**	**130,081**	**181,211**	**238,833**	**313,851**	**410,015**	**517,049**	**581,766**	**622,400**	**655,690**
Ending cash				**70,797**	**130,081**	**181,211**	**238,833**	**313,851**	**410,015**	**517,049**	**581,766**	**622,400**	**655,690**	**658,022**

Table 22-1: Summary of financial model (Page 2 of 4).

Cash flow Detail														
REVENUE			**Total (undiscounted)**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**
Price per lobe (USD $/ct)	**Annual Increase**	**# Yrs (escalation)**												
North	2.5%	4		222	228	233	239	245	245	245	245	245	245	245
Centre	2.5%	4		367	376	386	395	405	405	405	405	405	405	405
South	2.5%	4		716	734	752	771	790	790	790	790	790	790	790
Working Stockpile	2.5%	4		661	678	694	712	730	730	730	730	730	730	730
LOM Stockpile	2.5%	4		606	621	637	653	669	669	669	669	669	669	669
Average (calculated)	2.5%	4		520	639	521	663	712	747	782	705	754	764	585
Carats sold per lobe (ct)														
North			138,831	58,597	21,630	19,140	15,888	4,386	-	-	-	-	-	19,191
Centre			491,010	20,454	54,374	195,428	62,053	61,892	44,120	8,879	35,321	-	-	8,487
South			1,965,406	104,812	244,520	138,078	215,245	267,454	346,252	393,815	80,605	69,629	86,617	18,379
Working Stockpile			209,456	-	-	-	-	-	-	-	150,188	59,268	-	-
LOM			158,388								-	42,753	64,284	51,351
Total			**2,963,091**	**183,863**	**320,524**	**352,646**	**293,186**	**333,733**	**390,373**	**402,694**	**266,114**	**171,650**	**150,901**	**97,408**
Gross revenues (before royalties)														
North			31,970	13,008	4,922	4,464	3,798	1,075	-	-	-	-	-	4,703
Centre			192,128	7,507	20,454	75,353	24,524	25,072	17,873	3,597	14,309	-	-	3,438
South			1,522,323	75,045	179,453	103,869	165,966	211,377	273,653	311,244	63,704	55,030	68,456	14,525
Working Stockpile			152,823	-	-	-	-	-	-	-	109,581	43,243	-	-
LOM			112,446							-	-	31,193	46,903	34,349
Total			**2,011,690**	**95,560**	**204,829**	**183,686**	**194,289**	**237,524**	**291,527**	**314,841**	**187,593**	**129,466**	**115,359**	**57,016**

Table 22-1: Summary of financial model (Page 3 of 4).

PRODUCTION AND COST			Total (undiscounted)	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Mining (,000 tonnes)														
Ore mined			20,814	1,577	2,795	2,562	2,657	3,595	3,955	3,673	-	-	-	-
Waste mined - Cut 1			66	35	32	-	-	-	-	-	-	-	-	-
Waste mined - Cut 2			30,369	7,363	6,771	6,055	3,480	2,617	2,771	1,313	-	-	-	-
Total mining			51,249	8,975	9,597	8,617	6,136	6,212	6,726	4,986	0	0	0	0
Strip ratio			2.46	5.69	3.43	3.36	2.31	1.73	1.70	1.36				
			-											
Processing			-											
Plant Feed (tonnes)			-											
North			1,038,379	318,768	143,682	165,994	140,749	42,198	-	-	-	-	-	226,989
Centre			3,370,769	117,137	340,198	1,317,246	450,888	427,901	291,812	63,476	257,619	-	-	104,492
South			15,427,396	1,024,435	2,005,880	1,014,499	1,898,124	2,019,661	2,197,948	2,434,264	824,777	800,587	995,904	211,317
Working Stockpiles			2,103,027		-	-	-	-	-	-	1,407,364	695,663	-	-
LOM			3,464,518								-	993,510	1,493,856	977,152
Total			25,404,090	1,460,340	2,489,760	2,497,740	2,489,760	2,489,760	2,489,760	2,497,740	2,489,760	2,489,760	2,489,760	1,519,950
Plant Feed (grade)														
North				18.38	15.05	11.53	11.29	10.39	0.00	0.00	0.00	0.00	0.00	8.45
Centre				17.46	15.98	14.84	13.76	14.46	15.12	13.99	13.71	0.00	0.00	8.12
South				10.23	12.19	13.61	11.34	13.24	15.75	16.18	9.77	8.70	8.70	8.70
Working Stockpiles				0.00	0.00	0.00	0.00	0.00	0.00	0.00	10.67	8.52	0.00	0.00
LOM												4.30	4.30	5.26
Carats produced per lobe														
North			138,831	58,597	21,630	19,140	15,888	4,386	-	-	-	-	-	19,191
Centre			491,010	20,454	54,374	195,428	62,053	61,892	44,120	8,879	35,321	-	-	8,487
South			1,965,406	104,812	244,520	138,078	215,245	267,454	346,252	393,815	80,605	69,629	86,617	18,379
Working Stockpiles			209,456	-	-	-	-	-	-	-	150,188	59,268	-	-
LOM			158,388									42,753	64,284	51,351
Total			**2,963,091**	**183,863**	**320,524**	**352,646**	**293,186**	**333,733**	**390,373**	**402,694**	**266,114**	**171,650**	**150,901**	**97,408**
MCF/Recovery				100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Operating costs per tonne	**Annual Increase**	**# Yrs (escalation)**												
Ore mined	0.0%	20		3.09	3.16	3.23	3.40	3.60	3.93	4.51	0.00	0.00	0.00	0.00
Waste mined	0.0%	20		2.99	3.10	3.24	3.43	3.61	3.90	4.40	0.00	0.00	0.00	0.00
Ore Incremental Cost	0.0%	20		0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32
Processing costs	0.0%	20		10.62	10.62	10.62	10.62	10.62	10.62	10.62	10.62	10.62	10.62	10.62
Calculated costs of production														
Ore mined			76,057	4,877	8,836	8,270	9,021	12,944	15,556	16,553	-	-	-	-
Waste mined			100,848	22,132	21,103	19,638	11,944	9,446	10,807	5,779	-	-	-	-
Mining Contractors Fixed Manangment Fee			26,482	2,037	4,074	4,074	4,074	4,074	4,074	4,074				
Ore Incremental Mining Cost			6,660	504	894	820	850	1,150	1,266	1,175	-	-	-	-
Processed Costs			269,791	15,509	26,441	26,526	26,441	26,441	26,441	26,526	26,441	26,441	26,441	16,142
Site administration			88,058	5,527	11,055	11,055	11,055	11,055	11,055	11,055	4,051	4,051	4,051	4,051
Other			-											
Total production cash costs			**567,897**	**50,587**	**72,403**	**70,383**	**63,385**	**65,110**	**69,199**	**65,161**	**30,492**	**30,492**	**30,492**	**20,193**

Table 22-1: Summary of financial model (Page 4 of 4).

Taxes Payable				**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**
Less Capital Allowance				(4,750)	(9,500)	(9,500)	(9,500)	(9,500)	(9,500)	(9,500)	(9,500)	(9,500)	(14,500)	(23,695)
Less Interest														
Income before tax				27,175	95,198	78,592	94,930	131,296	174,783	199,360	121,925	70,715	53,286	2,990
Tax Calculation				**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Income bef. tax				27,175	95,198	78,592	94,930	131,296	174,783	199,360	121,925	70,715	53,286	2,990
Taxable Income				27,175	95,198	78,592	94,930	131,296	174,783	199,360	121,925	70,715	53,286	2,990
Tax Formula (see Section 4.2.2)														
a	**70**													
b	**1500**													
P/R = x (taxable income/gross revenues)				0.28	0.46	0.43	0.49	0.55	0.60	0.63	0.65	0.55	0.46	0.05
Other Tax														
Tax percentage 70-(1500/x)				**22**	**38**	**35**	**39**	**43**	**45**	**46**	**47**	**43**	**38**	**22**
Tax payable USD M			441,830	5,979	35,914	27,461	37,308	56,279	78,619	92,326	57,208	30,081	19,996	658

23. Adjacent properties

The information in this section was extracted and summarized from Oberholzer et al. (2017).

The Karowe Mine is based on the AK6 kimberlite pipe, which is part of the Orapa kimberlite field. Nine kimberlite pipes in this field are either operating mines or have been mined in the past. Current major adjacent diamond mines are shown in Figure 23-1 and summary details are provided in Table 23-1. Orapa is the second largest commercially exploited kimberlite in the world. The Letlhakane Mine produces diamonds of very high quality. The Damtshaa Mine is based on four relatively low-grade kimberlites.

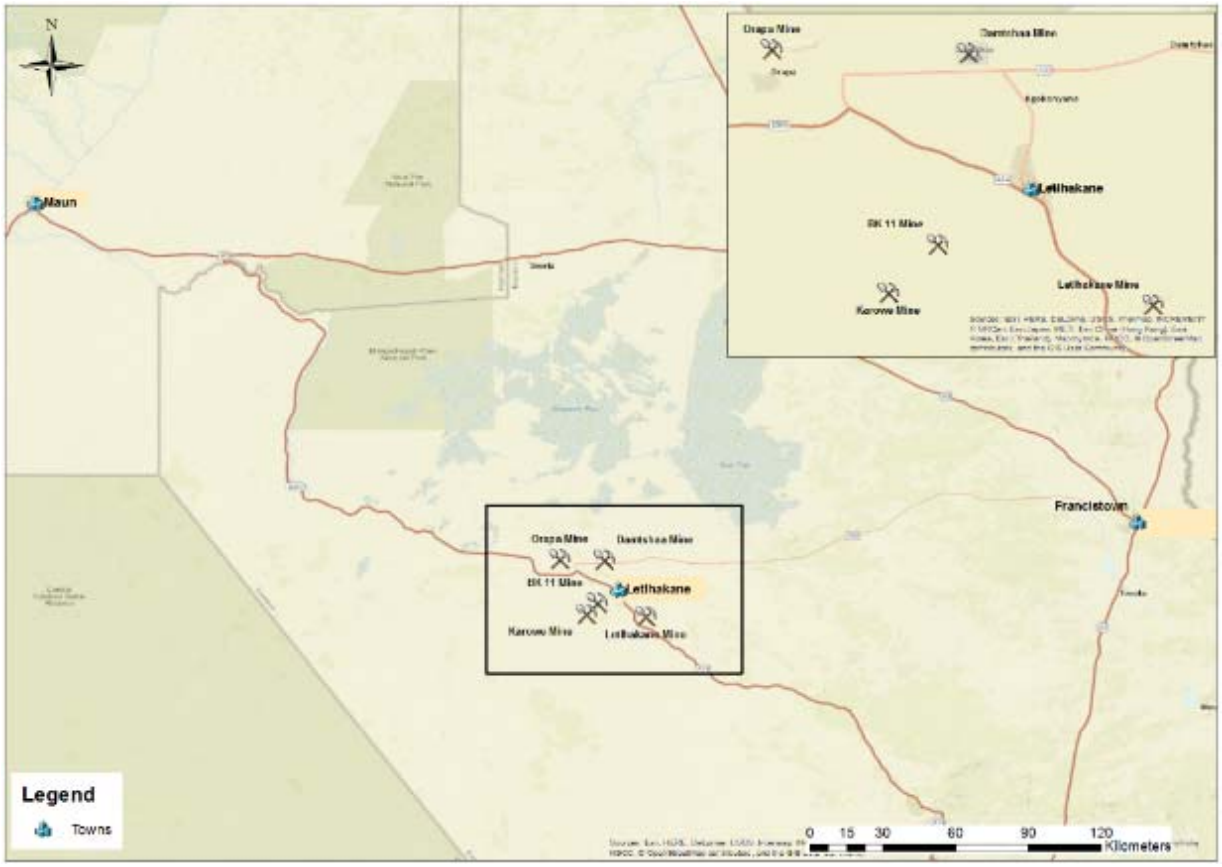

Figure 23-1: Locations of major diamond mines in close proximity to the Karowe Mine. Extracted from Oberholzer et al. (2017).

Table 23-1: Summary information for the nearby Orapa, Letlhakane and Damtshaa mines. All data sourced from the Anglo American Ore Reserves and Mineral Resources Report 2016.

Mine	Parameter	Description
Orapa	Owner	Debswana Diamond Mining Company (Pty) Ltd
	Mining Licence	Valid up to 2029
	Mining Started	1971
	Mining Method	Open Pit
	Grade	101.3 cpht (Measured and Indicated)
	Geology	Kimberlite AK/1
	Life of Mine	14 Years up to 2030
	Resource/Reserves	295.4 Mt (Measured and Indicated)
Letlhakane	Owner	Debswana Diamond Mining Company (Pty) Ltd
	Mining Licence	Up to 2029
	Mining Started	1977
	Mining Method	Open Pit
	Grade	31.7 cpht (Measured and Indicated)
	Geology	Kimberlite DK/1 and DK/2
	Life of Mine	1 Year up to 2017
	Resource/Reserves	22.2 Mt (Measured and Indicated)
Damtshaa	Owner	Damtshaa Mine
	Mining Licence	Up to 2029
	Mining Started	2002
	Mining Method	Open Pit
	Grade	25.0 cpht (Measured and Indicated)
	Geology	BK/9 and BK/12
	Life of Mine	18 Years up to 2034
	Resource/Reserves	4.4 Mt (Measured and Indicated), 19 Mt (Inferred)

24. Interpretation and conclusions

This Technical Report provides an update to the most recent previous AK6 Mineral Resource Estimate reported in Oberholzer et al. (2017). Evaluation work carried out in 2017 has increased confidence in the Mineral Resources present at depth. Indicated Mineral Resources now extend from surface to an elevation of 400 masl. Inferred Mineral Resources extend from 400 masl to the base of the current geological model at 256 masl. The basis for all estimates are described in Section 14 and an updated Mineral Resource statement (as of the end of December 2017) is provided in Section 14.7.

A revised Mineral Reserve Estimate (Table 15-1) has been generated from this updated Indicated Mineral Resource Estimate using the modifying factors discussed in Section 16. All Mineral Reserves are classified as Probable based on their Indicated Resource classification (Section 14.7). The Mineral Reserves reported are attributable solely to ore to be mined from the open pit at Karowe.

25. Recommendations

A microdiamond-based approach has been used to estimate grade in the deep part of AK6 below 604 masl (Section 14.4.6). This approach is based on the logging and sampling of 15 cores holes drilled at AK6 during

2017. These holes (pierce points and internal coverage) provide broadly accurate constraints on the volume and nature of kimberlite present to a depth of 400 masl and provided adequate material for bulk density / microdiamond sampling to support estimation of grade at a level of confidence suitable for reporting of Indicated Mineral Resources. However, due to gaps in the pierce point coverage, the exact location of the kimberlite/wall-rock contact is not well constrained in certain portions of the pipe and, to the extent that these need to be better constrained (e.g. for detailed planning for underground mining), further drilling will be required.

Pipe volume and the presence / extent of the M/PK(S) domain below 400 masl are not well constrained. It is considered likely that limited additional drilling (approximately 8 holes) planned to obtain pierce points and internal coverage in the 400 to 256 masl elevation range would provide sufficient constraints on this material to allow its classification at an Indicated level of confidence. This assumes that additional work does not result in the discovery of additional (previously unrecognised) geological units for which representative macrodiamond and value data are not available.

Additional geotechnical and hydrogeological drilling programmes should be undertaken along with detailed design studies (see recommendations from Oberholzer et al, 2017) to determine the potential for underground mining operations at Karowe. Conversion of the remaining Indicated Mineral Resources below the base of the Cut 2 Open Pit to a Mineral Reserve would be the ultimate goal of these additional studies.

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27. Authors

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28. Qualified persons certificates

CERTIFICATE of QUALIFIED PERSON: Tom E. Nowicki

I, Tom E. Nowicki, P.Geo., do hereby certify that:

1. I am a Senior Principal Geoscientist with Mineral Services Canada Inc. with an office at 501 – 88 Lonsdale Avenue, North Vancouver, BC, V7M 2E6, Canada.
2. I am a graduate of the University of Cape Town having obtained the degree of Bachelor of Science (Honours) in Geology in 1986 and Ph.D. Degree in geochemistry in 1998.
3. I am a graduate of Rhodes University (Grahamstown, South Africa) having obtained the degree of Masters of Science in Economic Geology in 1990.
4. 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.
5. I am a Registered Professional Geoscientist in good standing in British Columbia (P.Geo. registration number 30747).
6. This certificate applies to the Technical Report titled "2018 Independent Technical Report for the Karowe Mine: Updated Mineral Resource Estimate", with an effective date of 7 August 2018, (the "Technical Report") prepared for Lucara Diamond Corp. ("the Issuer").
7. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. 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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
9. I have read National Instrument 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. I am responsible for Sections 1.1 to 1.5, 1.7, 2 to 14 and 23 to 28 of the Technical Report.
11. I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
12. I visited the Karowe Diamond Mine on the 3rd and 4th July 2013.
13. My prior involvement with the Karowe Diamond Mine that is the subject of this Technical Report includes resource estimation work reported in Lynn, et al. (2014), and Oberholzer et al. (2017).

Effective Date: 7 August 2018

(original signed and sealed) "Tom E. Nowicki, P.Geo."

Tom E. Nowicki, P.Geo.

CERTIFICATE of QUALIFIED PERSON: John Armstrong

I, John Armstrong PhD, P.Geo. (NAPEG) of Vancouver, British Columbia, Canada do hereby certify that:

1. I am Vice President, Technical Services, for Lucara Diamond Corporation (2000-885 West Georgia Street, Vancouver, British Columbia, V6C 3E8, Canada) and a co-author of the "2018 Technical Report for the Karowe Mine: Updated Mineral Resource Estimate" (the Technical Report), dated 7 August 2018, and prepared for Lucara Diamond Corporation.
2. 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.
3. I am a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License # 1697).
4. 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.
5. I have visited the Karowe Diamond Mine on a regular basis since October 2013 with the most recent visit being October 2017.
6. I am responsible for Sections 16.1,16.2,17,18,19,20,21 and 22 of the Technical Report.
7. 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.
8. I have been involved with mining, production, and diamond sales activities at the Karowe Diamond Mine since October 2013.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with such instrument and form.
10. 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.

Effective Date: 7 August 2018

(original signed and sealed) "John Armstrong, P.Geo."

John Armstrong, P.Geo.

CERTIFICATE of QUALIFIED PERSON: Lofty Julius Hendrik Fourie

I, Lofty Julius Hendrik Fourie, do hereby certify that:

1. I am a director of Lofty Mining (Pty) Ltd with business address at 11 Siebolds Road, Darrenwood Ext 2, Randburg, Gauteng, South Africa, 2194;
2. This certificate applies to the Technical Report titled "2018 Technical Report for the Karowe Mine: Updated Mineral Resource Estimate", with an effective date of 7 August 2018, (the "Technical Report") prepared for Lucara Diamond Corporation ("the Issuer");
3. I am responsible for Sections 1.6, 15, 16.3 and 16.4 of the report;
4. I have not personally visited the Karowe Mine near Letlhakane in Botswana as part of the due diligence necessary to compile this report;
5. I hold a Bachelor of Engineering degree in Mining from the University of Pretoria, South Africa in 1980;
6. I am a Professional Engineer in good standing with the Engineering Council of South Africa (ECSA), registration number 870027;
7. I have worked as a mining engineer for a total of 35 years in a number of roles, including mining operations, mining projects and mining consulting over a wide range of commodities, mining methods and styles of mineralisation;
8. I am independent of the Issuer as defined in Section 1.5 of National Instrument 43-101;
9. I have no prior involvement with the Karowe Diamond Mine that is the subject of this Technical Report;
10. 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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
11. I have read National Instrument 43-101 and Form 43-101 F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. 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.

Effective Date: 7 August 2018

(original signed and sealed) "LJH Fourie (Pr Eng)."

LJH Fourie (Pr Eng)