LUCARA DIAMOND CORP.

KAROWE DIAMOND MINE

2023 FEASIBILITY STUDY TECHNICAL REPORT

LOCATION: BOTSWANA
EFFECTIVE DATE: JUNE 30, 2023
REPORT DATE: MARCH 12, 2024

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NOTICE

JDS Energy & Mining Inc. prepared this National Instrument 43-101 Technical Report, using the guidelines of Form 43-101F1, for Lucara Diamond Corp. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 EXECUTIVE SUMMARY

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Lucara Diamond Corp. (Lucara) to carry out an updated Feasibility Study (FS) for the Karowe Diamond Mine (KDM) Underground Mine Expansion Project (UGP or Project) currently being built to establish underground (UG) mining after the completion of the open pit (OP) mining. This technical report describes the combined life of mine (LOM) of the OP and UGP plans. All currency figures quoted in this report refer to United States (US) dollars (US$ or $) unless otherwise noted.

This report is updated from the original 2019 UGP FS and encompasses the following significant modifications:

- Advancement of detailed engineering designs;
- Re-modelling of the hydrogeological conditions;
- Modifications to the mine design;
- Re-baselining the UGP schedule and as a result, the OP mine and processing facility production plans;
- Re-estimation of the current operations budgets and the Project's capital and operating costs projections;
- Change to UG dewatering and grouting methodology;
- Changes to groundwater management on surface;
- Consideration of the Project construction progress (infrastructure and UG development) to the effective date of this report;
- Revised economic modelling with updated diamond prices and exchange rates, exclusion of sunk costs and inclusion of financing costs; and
- Revised waste management plans.

This report was prepared using guidance from the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101 and has an effective date of June 30, 2023.

For this updated FS, JDS was assisted by consultants and Qualified Persons (QPs) from the following independent companies:

- **DRA Botswana (Pty) Ltd.**: Mineral processing description;

- **Itasca Denver**: Hydrogeological modelling;
- **K-Met Consultants Inc.**: Metallurgical testing;
- **Knight Piésold (Pty) Ltd. (Botswana) (KP)**: Waste material management;
- **PRIZMA LLC**: Environment, permitting and social considerations;
- **SRK (South Africa)**: Geotechnical analysis, and
- **SRK Consulting (Canada) Inc.**: Geology, Mineral Resource estimation and UG material flow simulation.

1.2 Project Description

KDM is an existing OP mine and processing facility located in Central Botswana. The mine began commercial operations in July 2012 and currently operates at circa 2.7 million tonnes per annum (Mt/a) of feed to the processing plant. KDM has processed over 28 million tonnes (Mt) of ore and sold over 3.9 million carats (Mct) since the start of operations.

The mine has established itself as one the world's most prolific producers of large, gem quality, Type IIa diamonds. Since 2015, KDM has produced three diamonds greater than 1,000 ct in weight, and two of the world's most valuable rough diamonds: the 1,109 ct Lesedi La Rona ($53 M) and the 813 ct Constellation diamond ($63 M). Subsequent to the effective date of this report (June 30, 2023) KDM recovered its fourth +1,000 ct diamond, a 1,080 ct white gem. Roughly 70% of the mine's revenue is generated by +10.8 ct diamonds (Specials) that make up greater than 6% of the carats produced.

The in-situ OP reserve is planned to be fully depleted by 2025. The mine currently has approximately two years of stockpiled kimberlite ore. This FS evaluates extending the mine life by establishing UG mining production after depletion of the OP. Surface ore stockpiles are planned to bridge the production gap between the closing of the OP and the start of UG production. Stockpiles are also used opportunistically through the mine life to balance feed to the processing plant.

The UG expansion is summarized as follows:

- Mining:
  - Extraction of the South Lobe only as the extensions, at depth, of the North and Centre lobes are of insufficient tonnage and value to support UG mining below the OP;
  - Blind sinking an 8.5 metres (m) finished diameter Production Shaft (P/S) approximately 740 m deep equipped to hoist a nominal 7,400 tonnes per day (t/d) of ore and additional development waste;
  - Blind sinking a 6 m finished diameter unequipped Ventilation Shaft (V/S);

- Bulk stoping utilizing mass long hole shrinkage mining – a form of fully assisted “caving”;
- Hoisting of 37 Mt of UG ore mined at a grade of approximately 14.2 carats per hundred tonnes (cpht) providing 5.2 Mct recovered (UG only); and
- Extraction of approximately 400 m vertical of the South Lobe, of the AK06 kimberlite from 310 m above sea level (masl) (700 m below surface) to the bottom of the depleted OP (approximately 710 masl or 300 m below surface).

* Processing ore through the existing processing plant at a throughput of 2.7 Mt/a;
* An eight-year UG construction period beginning 2020 and ending in 2027; and
* 15 years of planned UG operations from 2028 through 2042.

1.3 Location, Access and Ownership

KDM encompasses approximately 1,523 hectare (ha) in the Central District of Botswana, 23 kilometers (km) west of the idle Letlhakane diamond mine and 25 km south of the operating Debswana Orapa diamond mine.

The geographic coordinates of KDM is 25° 28’ 13” E / 21° 30’ 35” S.

The mine is accessed via a well maintained, 15 km all-weather gravel road from the paved A14 Highway connecting Serowe to Orapa. Letlhakane is the closest village located at the junction of the mine road with the A14 Highway and can be accessed from the major cities of Gaborone and Francistown by paved roads. The closest airport that is serviced by limited commercial flights is in Francistown, approximately 200 km away or a 2.5 hour drive. Several international commercial flights per day, mainly from Johannesburg and Cape Town utilize the airport in Maun which is about 350 km (4 hour drive) from the Project. There is also an airstrip within the nearby Debswana controlled Orapa Township. KDM has its own operational 1,500 m gravel airstrip but does not support international flights at the time of this report.

Mineral Rights in the Republic of Botswana are held by the State. Commercial mining occurs under Mining Licenses issued by the Minister of Minerals, Energy & Water Resources. Lucara has a 100% interest in KDM through its indirect, wholly owned subsidiary Lucara Botswana Pty Limited (Lucara Botswana) and operates under Mining License 2008/6L.

1.4 History, Exploration and Drilling

The AK6 kimberlite pipe was discovered by De Beers in 1969. Since its discovery, there have been a multitude of exploration and resource / reserve definition programs completed on the property. The most significant programs are outlined in Table 1-1.

Table 1-1: Historical Exploration Programs

Program	Work Completed	Duration
Early Evaluation	5 x 12¼" large diameter drillholes totalling 679 m, 97 t bulk sample	2003 - 2005
	DMS and diamond recovery	
	Geophysical surveys	
Phase 1 Advanced Exploration	44 x 6½" percussion holes for delineation totalling 4,575 m	2005 - 2006
	12 x cored boreholes (NQ) as LDD pilots, totalling 2,980 m	
	17 x inclined boreholes (NQ) for delineation totalling 6,904 m	
	13 x 23" LDD totalling 3,699 m	
	DMS processing and diamond recovery from 1,775 t	
Phase 2 Advanced Exploration	11 x cored boreholes (NQ) as LDD pilots totalling 4,181 m	2006 - 2008
	29 x inclined boreholes (NQ) for delineation totalling 8,679 m	
	12 x 23" LDD totalling 4,265 m	
	Trench bulk sampling at surface	
	DMS processing and diamond recovery from 2,235 t	
Delineation and Geotechnical Drilling	15 x cored borehole (HQ and NQ) totalling 12,272 m	2016 - 2017
	916 microdiamond samples (7,315 kg)	
Delineation and Geotechnical Drilling	37 x cored boreholes (HQ and NQ) totalling 23,958 m	2018 - 2019
	153 microdiamond samples (1,232.8 kg)	
Shaft Investigation	2 x cored boreholes (NQ) totalling 1,514 m	2020 - 2021

Source: Lucara (2023)

1.5 Geology and Mineralization

KDM is exploiting the AK6 kimberlite which is part of the Orapa Kimberlite Field (OKF) in the Central District of Botswana. The OKF includes at least 83 kimberlite bodies of post-Karoo age. Three of these (AK1, BK9, and AK6) have been, or are currently being mined, and four (BK1, BK11, BK12 and BK15) are recognized as potentially economic deposits. KDM is one of the world's most significant producers of large and high-value diamonds including Type IIa and coloured diamonds.

The OKF lies on the northern edge of the Central Kalahari Karoo Basin where the Karoo succession dips very gently to the south-southwest and off-laps against Precambrian rocks that occur at shallow depth within the Makgadikgadi Depression. The country rock at KDM is sub-outcropping flood basalt of the Stormberg Lava Group (~130 m thick), underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup (~345 m thick), below which is the granitic basement.

AK6 is a roughly north-south trending elongate kimberlite body with a surface area of ~3.3 ha and maximum area of ~8 ha at approximately 120 m below surface. It comprises three geologically distinct, coalescing pipes known as the North, Centre and South Lobes that taper with depth into discrete roots. The kimberlite in each lobe is different, in terms of its textural characteristics, relative proportion of internal country rock dilution, degree of weathering and alteration, as well as the characteristics of mantle-derived components including the diamond populations. The South Lobe is the largest of the three lobes and is distinctly different from the North and Centre Lobes which are similar in terms of their geological characteristics. The South Lobe is broadly massive and more homogeneous than the North and Centre Lobes which exhibit greater textural complexity and more variable and higher proportions of internal country rock dilution.

The kimberlite in each lobe has been grouped into mappable units (Table 1-2) based on its geological characteristics and interpreted grade potential. Units occurring in more than one lobe (e.g., BBX, CKIMB, WK) were modelled as separate domains for each lobe (denoted by N, C or S suffix) in the geological model. The calcretized and weathered horizons in the upper portions of the lobes have now been mined out. Zones of high-country rock dilution (termed breccias) are present in all three lobes, and in the South Lobe these appear to be largely restricted to the upper now-depleted portion. The South Lobe additionally comprises two volumetrically dominant units, Magmatic / Pyroclastic Kimberlite (M/PK(S)) and Eastern Magmatic / Pyroclastic Kimberlite (EM/PK(S)), and six volumetrically minor units, one of which (KIMB3) becomes more prevalent with increasing depth in the pipe, particularly below 400 masl. M/PK(S) forms the dominant pipe infill above 600 masl, below which EM/PK(S) increases in volume at the expense of M/PK(S) to become the dominant infill below 500 masl. EM/PK(S) has now been drilled to 66 masl (~935 metres below surface (mbs)). The names applied to the two dominant units reflect the uncertainty historically regarding their textural classification (magmatic (M) or pyroclastic (P) kimberlite). The M/PK(S) and EM/PK(S) are broadly massive, olivine-rich and country rock xenolith-poor phlogopite monticellite kimberlites; they exhibit features suggesting they were formed extrusively and can be described as having clastogenic or apparent coherent texture (Scott Smith et al., 2017). The North and Centre Lobes are each infilled by single volumetrically dominant kimberlite units.

The current geological model (Figure 1-1) was first presented in Doerksen et al. (2019) as an update to the Nowicki et al. (2018) model based on the 2018/2019 FS drilling program and no additional updates have been made. The 2019 update involved revisions to the pipe margin to reflect mining gains in all three lobes, and changes to the pipe shell and internal domain model of the South Lobe based on 2018/2019 core drilling. The most significant changes were extension of the base of the model by 190 m (from 256 to 66 masl), reduction in the volume of M/PK(S) below 500 masl, and modelling of an additional internal domain encompassing the areas where drilling to date indicates KIMB3 is most prevalent. The pipe shells of the North and Centre Lobes were also updated based on the 2018/2019 core drilling.

The upper ~70 to 100 m of calcretized and weathered kimberlite and country rock breccia units, which are now mined out, are shown in a single colour to simplify Figure 1-1. Some domains are rendered transparent to display the internal domains.

Table 1-2: Kimberlite Units Identified in the AK6 Kimberlite

Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
	CKIMB	CKIMB(N)	Calcretized 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
Centre	BBX	BBX(C)	Country rock breccia
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
	CKIMB	CKIMB(C)	Calcretized 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)	Calcretized country rock breccia
	CKIMB	CKIMB(S)	Calcretized 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	KIMB3	Minor hypabyssal kimberlite; increasing volume below 500 masl
	KIMB4a	EM/PK(S)	Localized variant of EM/PK(S)
	KIMB5*	n/a	Volumetrically minor hypabyssal kimberlite
	KIMB6*	n/a	Volumetrically minor hypabyssal kimberlite
	KIMB7*	n/a	Volumetrically minor kimberlite

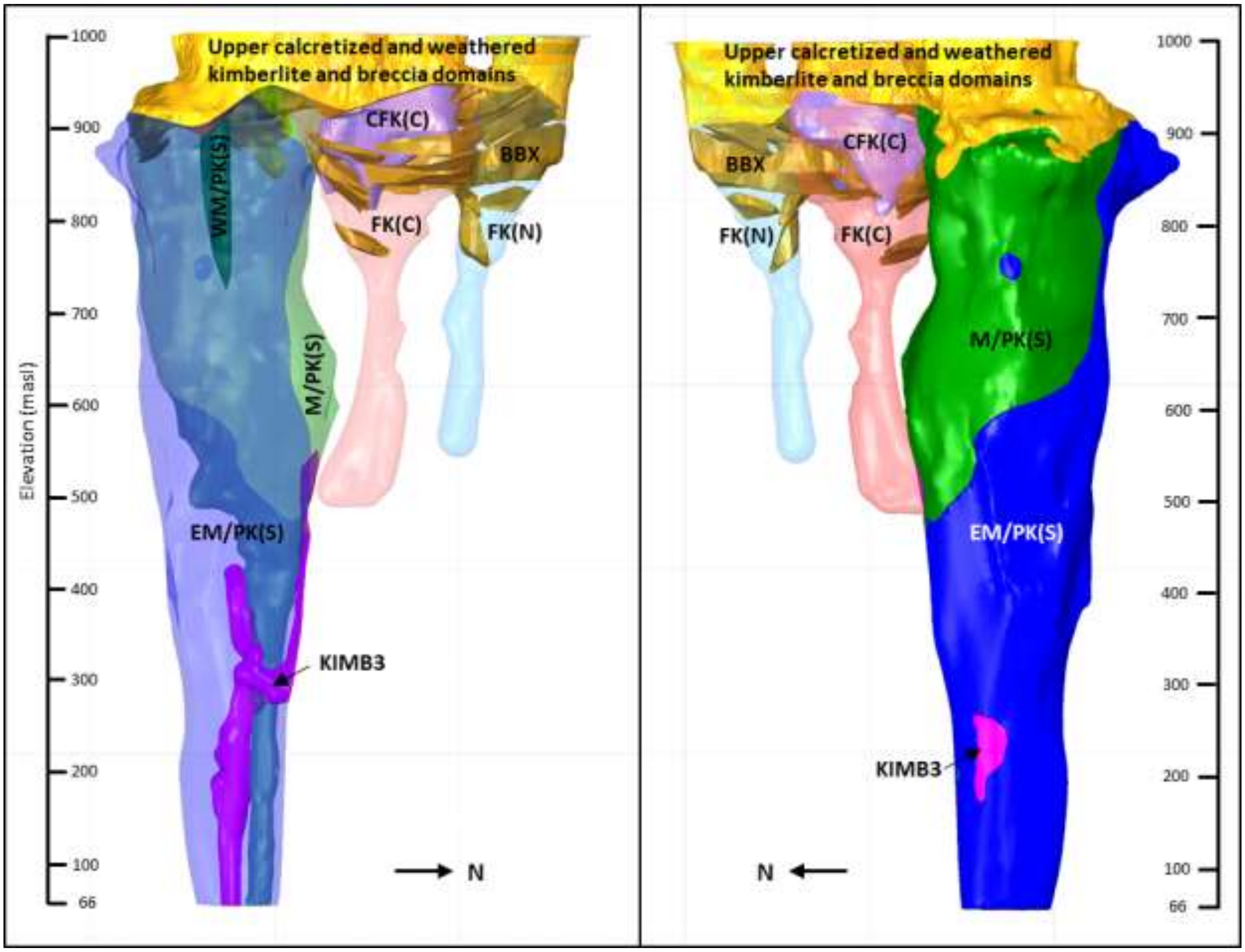
Notes:

*Minor units are included in the major domain models; same applies to KIMB3 intersections not included in the KIMB3 domain.

Units occurring in more than one lobe (e.g., BBX, CKIMB, WK) are modelled as separate domains for each lobe (denoted by N, C or S suffix) in the geological model.

Source: SRK (2023)

Figure 1-1: Internal Geological Domains of the AK6 Kimberlite



Source: SRK (2023)

1.6 Mineral Processing Testwork

An assessment of the plant capacity when treating UG ore was conducted by testing X-ray transmission sorting and milling performance of deeper UG ore.

1.6.1 Comminution Testwork

Comminution testwork to determine the characteristics of the deeper kimberlite ore was carried out at Base Metallurgical Laboratories (BaseMet) in Kamloops, BC, Canada in 2019. Bulk samples and HQ drill core representing EM/PK(S) and M/PK(S) zones of the South Lobe were collected from various depths throughout the deposit. Bulk samples were taken from the 2019

OP at approximately 900 masl. Diamond drill core was sampled at varying depths below the OP and within the planned UG mining area of the deposit. The testwork was completed to compare the hardness of EM/PK(S) and M/PK(S) samples and predict the effect on the existing Autogenous Grinding (AG) Mill with respect to the impact on production rate when the deeper UG material is processed.

The comminution testwork completed on the bulk samples included: Crushing Work Index (CWi), Bond Rod Mill Work Index (RWi), Bond Ball Mill Work Index (BWi) and JK Drop Weight. The HQ drill core testwork included RWi, BWi and SAG Mill Comminution (SMC).

The results of the samples tested indicate that there is not a significant difference in the hardness between EM/PK(S) and M/PK(S). The samples tested demonstrated similar characteristics to the material processed in the existing AG mill, and therefore, the UG material planned to be mined can be processed in the current comminution circuit at the planned production rate.

1.6.2 XRT Testwork

The predominant diamond separation and extraction process in the current process plant uses Tomra X-ray Transmission (XRT) bulk sorting machines to separate liberated diamonds from sized run of mine kimberlite and waste host rock. The XRT units are able to analyze the atomic density of materials and then physically separate the materials with a diamond / carbon signature from non-diamondiferous material.

The UG mine is planned to mine kimberlite through a carbonaceous shale host lithology. It is expected that some carbonaceous shale will report to the mill and potentially the XRT bulk sorters as dilution during the later stages of UG mining. The carbonaceous shales contain small lenses of coal which could potentially be recovered by the XRT units since both diamonds and coal are composed of carbon.

To test the ability of the XRT to differentiate and separate, coal, carbonaceous shale and other host rock lithologies from diamonds, samples of South Lobe kimberlite and waste host rock were sampled and shipped to Tomra’s laboratory in Germany.

The results of the tests determined that the coal and carbonaceous shales, as well as all other host waste rock lithologies could be identified and separated by the XRT machines from the diamonds and that the current Tomra system at the mine is suitable for the proposed UG ore.

1.7 Mineral Resource Estimate

The 2023 Mineral Resource Estimate for KDM incorporates drilling and sampling data obtained prior to 2018, and additional drilling and sampling information obtained in 2018/2019 which targeted delineation of the deep extension of South Lobe (deeper than ~600 m from surface). In 2019, the geological data were used to develop an updated internal geology model for the South Lobe and to update the external contacts for the North, Centre and South Lobes. The 2023 update also includes geological information and production data derived from OP mining to the end of June 30, 2023.

The internal geology of the South Lobe is comprised of two dominant domains, identified as the M/PK(S) and EM/PK(S) domains. A single diamond size frequency distribution (SFD) and diamond value model were used prior to 2019 to evaluate the South Lobe because OP production was strongly dominated by M/PK(S) material. Incremental OP production of EM/PK(S) material was initiated in early 2018 and sufficient data has since been amassed so that distinct SFD and diamond value distribution models are now defined for both the M/PK(S) and EM/PK(S) domains in the 2023 Mineral Resource update.

Value distribution models and estimates of average price per carat (US$/ct) for each kimberlite domain and lobe have an LOM production and sales information to the end of June 2023. The diamond value estimates incorporate current trends observed through diamond tenders, Clara and HB Antwerp sales data along with production data from KDM and are representative of the current status of the diamond market at the effective date. The value models exclude all revenue generated from diamonds sold for more than $10 M each since 2014, no escalation is applied to the diamond price assumptions.

The 2023 Mineral Resources for KDM, as summarized in Table 1-3, have been classified as either Indicated or Inferred Mineral Resources, according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). Mineral Resources reported are inclusive of those portions of the Mineral Resource that have been converted to Mineral Reserves and have an effective date of June 30, 2023.

Table 1-3: KDM 2023 Mineral Resource Statement (effective date of June 30, 2023)

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mcts)	Grade (cpht)	Average ($/ct)
Indicated	South_M/PK(S)	7.02	20.92	2.96	2.27	10.8	$707
	South_EM/PK(S)	6.77	19.77	2.90	4.16	21.0	$828
	Centre	0.30	0.81	2.57	0.12	15.5	$392
	North	0.18	0.42	2.45	0.05	11.6	$273
Total Indicated		**14.27**	**41.92**	**2.90**	**6.60**	**15.8**	**$793**
Inferred	South_M/PK(S)	0.10	0.31	3.05	0.03	10.5	$707
	South_EM/PK(S)	1.40	4.18	2.97	0.87	20.9	$828
	South_KIMB3	0.32	0.94	2.94	0.10	10.9	$707
Total Inferred		**1.82**	**5.42**	**2.97**	**1.01**	**18.6**	**$804**

Notes:

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. All numbers have been rounded to reflect accuracy of the estimate;
2. Mineral Resources are in-situ Mineral Resources and are inclusive of in-situ Mineral Reserves;
3. The base of the South Lobe Indicated Mineral Resource is 250 masl and 60 masl for the Inferred Resource;
4. Mineral Resources are exclusive of all mine stockpile material;
5. Mineral Resources are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond losses within the smaller sieve classes expected within the current configuration of the KDM process plant;
6. Inferred Mineral Resources are estimated on the basis of limited geological evidence and sampling, sufficient to imply but not verify geological grade and continuity. They have a lower level of confidence than that applied to an Indicated Mineral Resource and cannot be directly converted into a Mineral Reserve;

7. Average diamond value estimates are based on 2023 diamond sales data provided by Lucara Diamond Corp.; and
8. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery.

Source: SRK (2023)

1.8 Mineral Reserve Estimate

A consolidated OP and UG mine plan was developed to extract the economic portions of the KDM Indicated Mineral Resources plus stockpiled ore. The mine plan includes extraction of three adjacent lobes of kimberlite. The South Lobe is planned to be mined through a combination of OP and UG mining methods. The Centre Lobe is planned for extraction by OP mining methods only. The remaining North Lobe mined from the OP, is not considered a reserve. All Mineral Reserves are classified as Probable Reserves.

OP and UG design, schedule, and reserves estimates were prepared by JDS. Stockpile quantities were prepared by Lucara and reviewed by JDS and are included in the Mineral Reserve Estimate. A consolidated summary of the Mineral Reserve Estimate, by mining method and pipe, is presented in Table 1-4.

The effective date for the Mineral Reserve Estimate contained in this report is June 30, 2023 and was prepared by Qualified Person (QP) Brandon Chambers, P.Eng. All Mineral Reserves in Table 1-4 are classified as Probable Mineral Reserves. The Mineral Reserves, except stockpiles, are not in addition to the Mineral Resources, but are a subset thereof.

The QP has not identified any legal, political, or environmental risks that would materially affect potential Mineral Reserves development.

Table 1-4: KDM Mineral Reserve Estimate

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	LOM Diamond Price
		(Mt)	('000s ct)	(cpht)	($/ct)
Open Pit					
Centre	Probable	0.6	96	16.3	392
South - EM/PK(s)	Probable	1.3	323	25.4	828
South - M/PK(s)	Probable	3.6	384	10.7	707
Open Pit	**Total**	**5.5**	**803**	**14.7**	**718**
UG					
South - EM/PK(s)	Probable	18.6	3,361	18.1	828
South - M/PK(s)	Probable	18.4	1,871	10.2	707
UG	**Total**	**37.0**	**5,232**	**14.2**	**785**

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	LOM Diamond Price
		(Mt)	('000s ct)	(cpht)	($/ct)
Stockpile					
Mixed Stockpile	Probable	4.0	502	12.7	433
Life of Mine	Probable	5.8	296	5.1	574
Stockpile	**Total**	**9.7**	**798**	**8.2**	**485**
Combined					
All	**Total**	**52.2**	**6,834**	**13.1**	**742**

Notes:

1. Prepared by Brandon Chambers, P.Eng. JDS Energy & Mining Inc.;
2. CIM definitions were followed for Mineral Reserves;
3. Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the Mineral Resource block model assumptions and, therefore, have taken recoveries into account;
4. The bottom elevation of the Probable Reserve is 310 masl;
5. Mineral Reserves are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond loses within the smaller sieve classes expected within the current configuration of the KDM Process Plan;
6. Diamond price estimates are provided by Lucara; prices were derived from historical sales and adjusted for current market conditions;
7. Tonnages are rounded to the nearest 100,000 t, diamond grades are rounded to one decimal place to properly reflect the Reserve estimate accuracy;
8. Tonnage and grade measurements are in metric units and contained diamonds are reported as thousands of carats;
9. OP Mineral Reserves are estimated at a cut-off value of $37/t based on an OP mining cost of $13/t, a processing cost of $12/t and a G&A cost of $12/t;
10. UG Mineral Reserves are estimated at a cut-off value of $35/t based on a UG mining cost of $11/t, a processing cost of $12/t and a G&A cost of $12/t;
11. Mine Call Factor is a modifying factor used by Lucara which tracks the reconciliation between the block model and actual recovered carats. Mine Call Factor is assumed to be 100%, historically, this factor has reconciled either near or above 100%, however, in the 12-month period prior to the Reserve Statement, the Mine Call Factor has deviated away from historical average performance and is currently at 95%;
12. UG dilution assumptions in the 2019 FS were revised in 2023. UG dilution included in the Reserve was estimated from the following three sources:
    - 1.0 m of zero-grade overbreak from stoping adjacent to the granite host rock;
    - 2.7 Mt of zero-grade overbreak from stoping adjacent to sedimentary rocks (based on geomechanical modelling); and
    - Inclusion of inferred KIMB3 kimberlite within the overall pipe shape as zero-grade waste.
13. Stockpile Mineral Reserves are estimated at a cut-off value of $19/t based on a rehandle cost of $2/t, a processing cost of $12/t and a G&A cost of $5/t, when processed at the end of mine life;
14. Stockpile Reserves are not included in the KDM Mineral Resource Estimate, which covered only in-situ mineralized material;
15. Stockpile Reserves are based on surveyed volumes and block model grades; and
16. Stockpile LOM diamond price is determined from the weighted average of the North, Centre, South - M/PK(s), and South - EM/PK(s) lobe ratios.

1.9 Geotechnical and Hydrogeological Context

1.9.1 Geotechnical

The granite host rock and kimberlite ore are generally good quality with low weathering susceptibility, where most of the UG excavations will be sited. The contact zone has a higher joint frequency and increased clay content within the kimberlite. There are also a few weaker layers in the country rock (Ntane sandstones, Mosolatane red mudstone, Tlapana carbonaceous mudstones and weathered granite) and some of these layers are less resistant to weathering.

Due to the limited hydraulic radius of the pipe and relatively competent kimberlite, natural caving was considered unlikely. FLAC3D modelling showed that continuous caving would not occur and that the selected pyramidal sequence limits overbreak promote stability of the crown and sill pillars. The stresses induced on the drifts and drilling horizon are not anticipated to induce problematic closure. Modelling indicates that the infrastructure is not likely to be significantly influenced by subsidence and relaxation.

Monitoring of the blastholes and the cavity will be important to verify the performance of the excavation. Good draw control is essential for minimizing dilution from country rock and potential mud rushes throughout the life of the mine. Breakback monitoring in the rim tunnels and access drives on all levels is essential.

1.9.2 Hydrogeological

The OP operation is currently within the Ntane and Mosoltane sandstones. Dewatering and depressurization are critical in reducing the inflow to the pit and the pore pressure of the pit wall and pit bottom. These dewatering and depressurization measures will continue to the end of UG mining.

The UG mining will start in the granite. Because of the separation between the sandstone units and Mea/granite units by ~200 m thick mudstone/shale, the dewatering of the OP has essentially no impact on the groundwater condition of the Mea/granite units. The high pressure and possibly permeable Mea/granite units could lead to as high as 12,000 cubic metres per day (m^3/day) of inflow rate to the UG workings, however, packer test results and drill hole observations through the Mea have shown variable results and inflows could be lower.

The design of the UG drainage gallery that targets the kimberlite contact zone is a practical measure to control the flow in the mining zones. However, given the lack of hydrogeologic data in the Mea/granite units and the assumed highly permeable fracture corridor, there are uncertainties in the predictive inflow to the UG mine workings. The hydraulic investigation/monitoring should be planned and commenced as soon as UG access becomes available.

1.10 Mining

KDM is an existing OP operation, which has been in production since 2012. Conventional OP drill and blast mining with diesel excavators and trucks provide an average annual 2.7 Mt of kimberlite feed to the mill. All OP mining activities are performed by Botswanan mine contractors working 365 days per year on three, eight-hour shifts in the pit. Lucara operates the processing facility with two, 12-hour shifts. The OP mine operation is expected to terminate mid-2025, ending at an elevation of approximately 713 masl.

There are substantial resources remaining below the economic extents of the OP that may be extracted by UG mine methods. A 7,400 t/d shaft operation utilizing long hole shrinkage mining (a form of fully assisted caving) is under construction to provide an additional 13 years of mine life to the KDM operation after an eight-year construction period which commenced in 2020.

The KDM resource is hosted by three distinct coalescing pipes, referred to as the North, Centre, and South Lobe. All lobes were sub-cropping, dip vertically, and vary in diameter and depth. The South Lobe is the most volumetrically significant of the three, and its Indicated Resources extend approximately 760 mbs (from 1,010 masl to 250 masl). The North and Centre Lobes extend below the OP limit but have been excluded from the planned UG mine as they are an Inferred Resource at depth.

The South Lobe contains four distinct domains, each with unique mineral properties. These domains are summarized as EM/PK(S), M/PK(S), KIMB3, and weathered kimberlite. Weathered kimberlite has been mined out by the OP and is no longer present in the Mineral Resource. KIMB3 is an Inferred Resource that has been, for reporting and economic modelling purposes, treated as zero-grade dilution in the UG mine plan. EM/PK(S) and M/PK(S) are the two economic mineralized domains within the South Lobe on which the UG mine plan is focused. The M/PK(S) domain is situated near surface and has approximately half the diamond grade and contained value of the lower EM/PK(S) domain. This geologic feature drives several mine plan design decisions which focus on accessing the deeper, higher-value EM/PK(S) resource early in the mine life.

The small hydraulic radius at depth (27 m), low in-situ (horizontal) stress in combination with high compressive and tensile strength of the kimberlite suggests that the resource will not cave naturally or with pre-conditioning and will, therefore, require drill and blast assistance. The resource economically favours a bottom up mine approach, which takes advantage of the higher value EM/PK(S) kimberlite at depth.

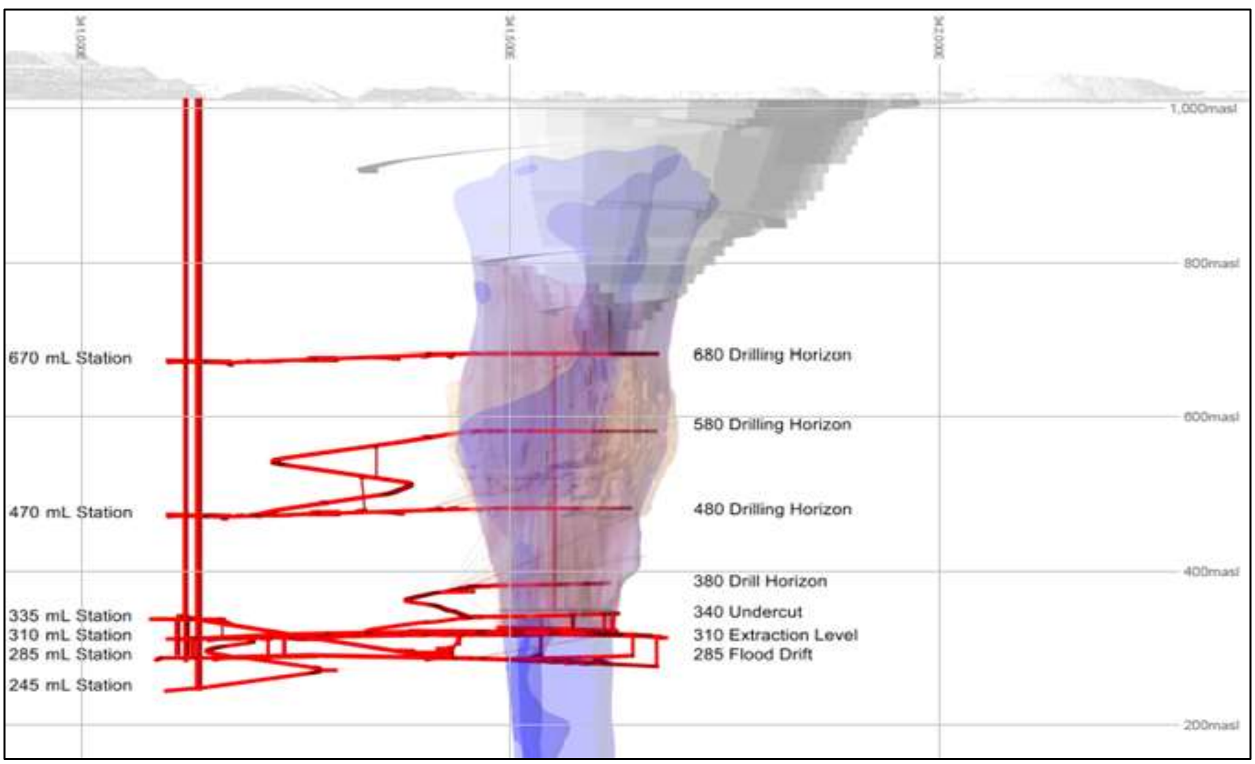
Long Hole Shrinkage (LHS) stoping is planned to systematically drill and blast the entire lobe on a vertical retreat basis. The method can be thought of conceptually as a fully assisted cave. In LHS, the blasted muck is left in the excavation during stoping to stabilize the host rock with only the swell extracted or pulled during the drill and blast phase. Mucking takes place from drawpoints at the bottom of the mine on the 310 Level (L) (310 masl). As ore is blasted, it swells beyond its in-situ volume, and this volume is mucked or pulled from the drawpoints to maintain a blasting void within the excavation. Once the ore is fully blasted to the bottom of the OP, the South Lobe is drawn empty by mucking the drawpoints.

Access to the UG mine will be from a 767 m deep P/S, 8.5 m in diameter, sunk from surface to 245 masl. The shaft will be equipped with two 21-t skips for production hoisting, a service cage for man and material movement, and an auxiliary cage for shaft inspections and personnel transport. Shaft conveyances will be managed by three independently operated winders, of which one currently exists on site and is performing shaft sinking duties. This shaft will also serve as the main fresh air intake to the mine. A second shaft, 6.0 m in diameter, 727 m deep, driven from surface to 285 masl, will serve as the main exhaust route and emergency egress for the mine. The two shafts are offset from the kimberlite pipe ~375 m northwest of the South Lobe, well outside of the potential subsidence zone, and 100 m from each other. Shafts will be driven blind using conventional drill and blast equipment and are being developed concurrently. Average sinking rates range from 1.9 to 2.4 metres per day m/day during steady state sinking in good ground. It is expected to take approximately six years to fully sink and equip both shafts, plus another two years to complete all UG development, capital installations, and production ramp up.

There will be a total of eight working levels in the mine, six of which will be accessed by a shaft station. Levels are named by their elevation in masl. The 310 L will serve as the primary working level and provide access to the main UG infrastructure including production drawpoints, crusher, and maintenance facilities. Above this level will be four drilling horizons: 380 L, 480 L, 580 L, and 680 L; where production equipment will work to drill and blast stopes. Other stations will serve as support services for ore handling and access to the shaft bottom.

Figure 1-2 shows an isometric view of mine development.

Figure 1-2: Mine Development Schematic



Source: JDS (2023)

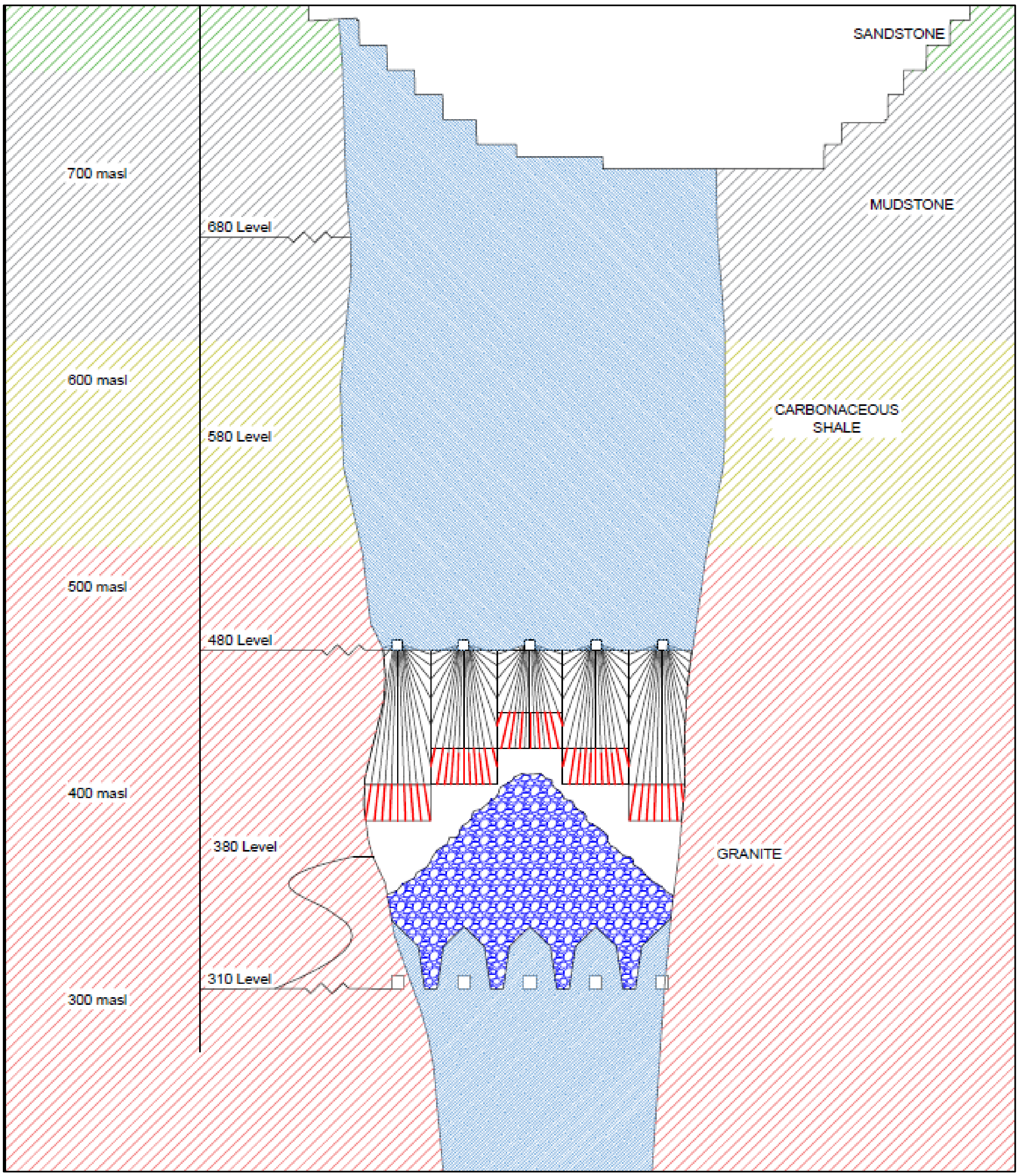
The UG lateral development will be driven by four development jumbos, initially mobilized to the 285 L and 310 L. Each crew will drive an average of 3.5 m/day in a priority heading and 2.75 m/day in a secondary heading, to a maximum of 11 m/day per working jumbo. After the majority of the development is complete on the 310 L, one jumbo will be sent up to the 480 L and another up to the 680 L. The last jumbo will remain on the 310 L for any rehabilitation work that will need to be completed throughout the mine life. During pre-production, a total of 16 km of development will be driven.

Drill horizons are spaced at 100 m vertical intervals to accommodate the in the hole hammer (ITH) drill's effective drill length of a 150 millimeter (mm) diameter hole. Drilling of the stopes will be completed by mainly down holes on a 4.35 m burden by 5.00 m spacing ring pattern. The average length of hole per ring will be 58 m, with an average 34 tonnes per metre (t/m) drilled. Stope production blasting will utilize a powder factor of 0.6 kilograms per tonne (kg/t) below the first drill horizon to ensure high rock fragmentation at the start of the shrinkage process. In the upper levels the powder factor will be reduced to 0.4 kg/t to match that of current OP operations which produces excellent fragmentation.

A pyramidal sequence is proposed for the drilling and blasting of the stopes at KDM. This blasting sequence will create a dome shape at the top of the blasted volume to maintain stability of the stope back. Stopes will be blasted sequentially upwards in 17.5 m increments until a 30 m sill pillar is left between the drill panel and the stope back. A final 30 m blast will wreck this sill pillar and terminate access to the drill panel at that location. The drill will relocate to the next above drill horizon and repeat the process until the lobe is fully blasted.

Figure 1-3 illustrates a schematic cross section of the pipe, showing the pyramidal advance of stopes.

Figure 1-3: Mining Method Illustration

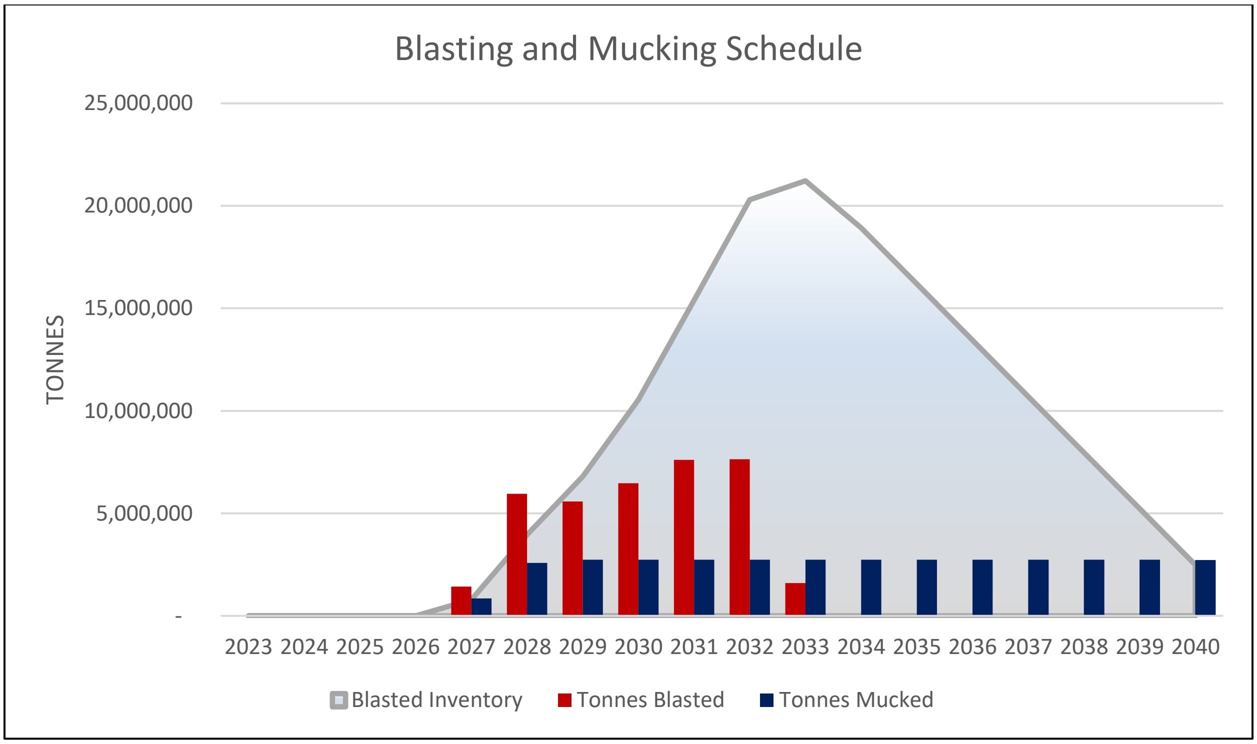


Source: JDS (2019)

Five ITH drills will be utilized to drill and blast approximately 21,000 t/d in order to supply 7,400 t/d of swell to the draw bells for the first six years of operations. Peak broken inventory occurs in year six for a total of 21 Mt. After six years, the South Lobe will be fully blasted, and mucking will continue at a constant rate of 7,400 t/d until the UG reserves are depleted at the end of year thirteen. Final operation will be spent processing low grade OP stockpile while concurrently performing site closure of the OP and UG mine.

The UG blasting and mucking schedule is outlined in Figure 1-4.

Figure 1-4: Blasting and Mucking Schedule



Source: JDS (2023)

The extraction level will be made up of five panels that are driven 31.5 m apart and run the entire length of the lobe. Each panel will access one of 50 drawpoints driven 18 m x 12 m in an offset herringbone pattern. The extraction level will contain one perimeter drive to allow traffic to go around panels in the event of a blockage or maintenance at the drawpoints. At the northwest side of the extraction level, the five panels will access a static grizzly tip from three sides. Re-muck bays will be located near the grizzly tip to allow for continued drawpoint mucking during comminution circuit maintenance and a quick re-handle once the circuit returns to normal operation. Three 21-t loaders will be required to maintain production at the draw bells. In addition, development loaders will remain on site following completion of the capital development campaign to assist with mucking during periods of re-handle or increased haul distances due to panel rehabilitation.

Material dumped onto the grizzly will feed a 1.3 m x 1.5 m (50” x 60”) UG jaw crusher with 960 tonnes per hour (t/h) capacity located 32 m below the extraction level. Crushed material will report to a sacrificial conveyor equipped with metal detectors and magnets. This material will be transferred to a longer conveyor for transport to the 335 L shaft station and onto a reversible transfer conveyor for discharge into one of two fine ore storage bins, each with a capacity of 2,400 t.

The storage bins will discharge onto a skip loadout conveyor which will direct material to one of two 21-t skips. Skips will cycle to surface every two minutes and dump into an elevated bin for either direct truck loading or for conveyance to a surface stockpile for rehandle. 39-t trucks will load at the shaft and tram ore to the plant or waste to the waste rock storage facility, some two kilometres away.

Table 1-5 states the annual schedule of material hoisted to surface from the UG operation.

Table 1-5: UG Production Schedule

Year	EM/PK(S)			M/PK(S)			Total		
	Tonnes	Grade	Carats	Tonnes	Grade	Carats	Tonnes	Grade	Carats
	Mt	cpht	Kc	Mt	cpht	kc	Mt	cpht	kc
2026	0.1	16.0	18	-	9.3	-	0.1	15.8	18
2027	1.1	18.6	208	0.1	9.8	13	1.2	17.7	221
2028	2.4	19.7	473	0.3	9.6	33	2.7	18.4	505
2029	2.4	19.9	486	0.3	10.0	29	2.7	18.8	515
2030	2.2	20.0	436	0.6	9.9	55	2.7	18.0	491
2031	1.9	18.6	346	0.9	9.6	84	2.7	15.7	431
2032	1.6	17.1	270	1.2	9.4	111	2.7	13.9	380
2033	1.9	13.7	256	0.9	9.0	79	2.7	12.2	334
2034	0.7	15.1	106	2.0	10.0	204	2.7	11.3	310
2035	1.0	12.2	120	1.8	10.4	182	2.7	11.0	302
2036	0.9	14.4	129	1.9	11.0	204	2.7	12.1	332
2037	1.0	19.9	201	1.7	11.0	190	2.7	14.3	391
2038	0.4	21.0	94	2.3	9.7	222	2.7	11.6	316
2039	1.0	22.6	217	1.8	10.0	179	2.7	14.5	396
2040	-	22.9	1	2.7	10.6	288	2.7	10.6	289
Total	**18.6**	**18.1**	**3,361**	**18.4**	**10.2**	**1,871**	**37.0**	**14.2**	**5,232**

Source: JDS (2023)

The ventilation network will consist of 460 cubic metre per second (m^3/s) fresh air intake by the P/S and two exhaust routes by the V/S and an in-pit raise for 366 m^3/s and 105 m^3/s respectively. Fresh air will be pulled into the mine workings through the P/S. Primary ventilation fans will consist of a bifurcated surface fan arrangement over the V/S collar, as well as a twin booster ventilation bulkhead at the base of the in-pit raise. UG, level ventilation will be controlled by a combination of regulators, doors, ducting, and auxiliary fans.

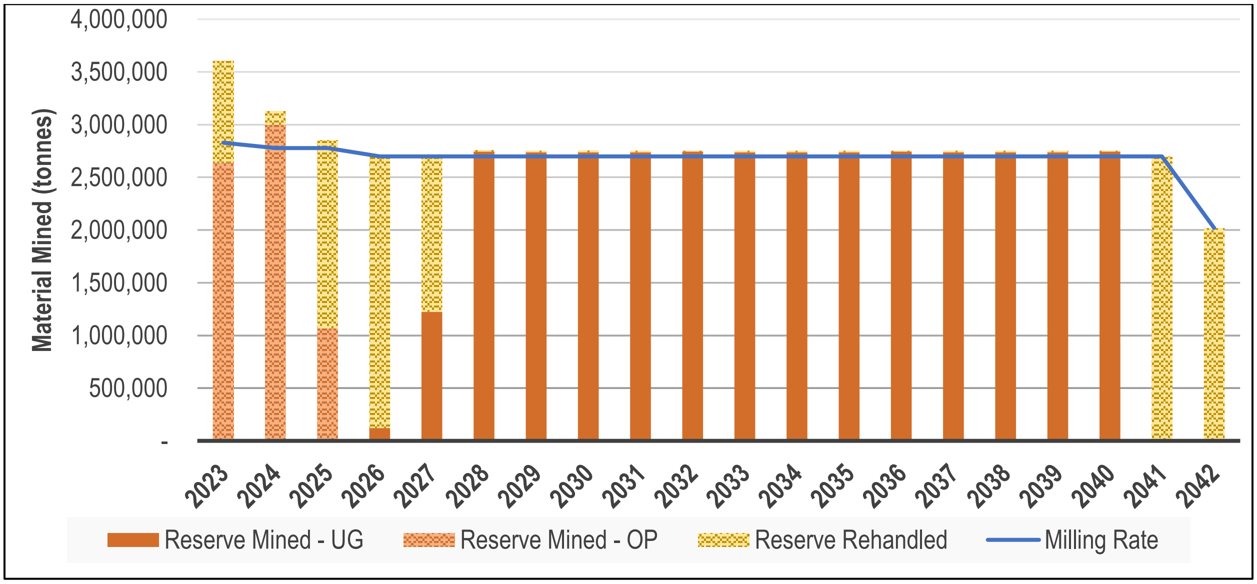
UG wet-bulb temperatures (WBT) will be maintained below 27.5 degrees Celsius (°C) by employing 7.5 Mega Watts of Refrigeration (MWR) through a surface bulk air cooler plant for eight months of the year. During the four cooler months of the year, May through August, mine air cooling will not be required.

Mine and ground water will be collected at the various level sumps and allowed to drain down via gravity to the main pump stations placed at strategic locations in the mine. Pump stations have been designed for a peak dewatering requirement of 500 cubic metre per hour (m^3/hr). To mitigate sudden inrushes of stormwater during major events, dedicated flood chambers will be provisioned below the extraction drive.

The UG mine will be contract developed and Owner operated. Contractors will be utilized for shaft sinking, lateral development, production drill and blast, and raise development. Applicable existing OP employees will be trained during pre-production to transition to the UG mine as the OP winds down and UG production ramps up. Total mine construction workforce required per day (day shift + night shift) will peak during pre-production at 550 persons.

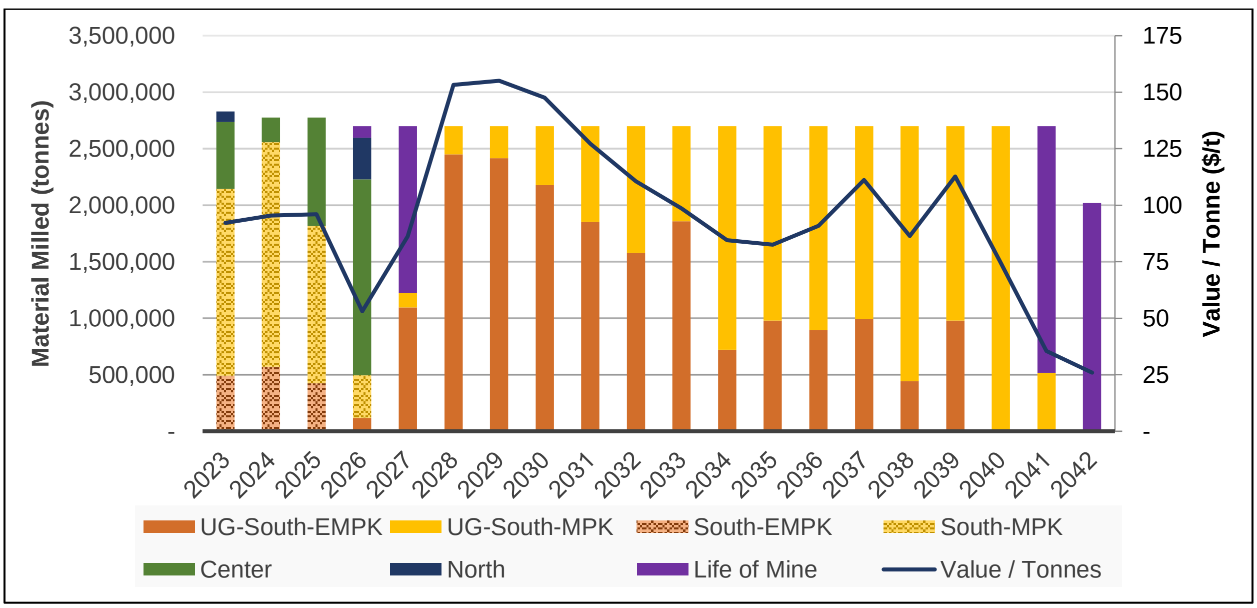
The OP will continue to operate until mid-2025. During the OP / UG transition, surface stockpiles will be consumed by the plant based on processing the highest value ore first. The total blended mine and mill feed from both UG, OP, and stockpile operations is shown in Figure 1-5 through Figure 1-7.

Figure 1-5: Summary of Mine Production



Source: JDS (2023)

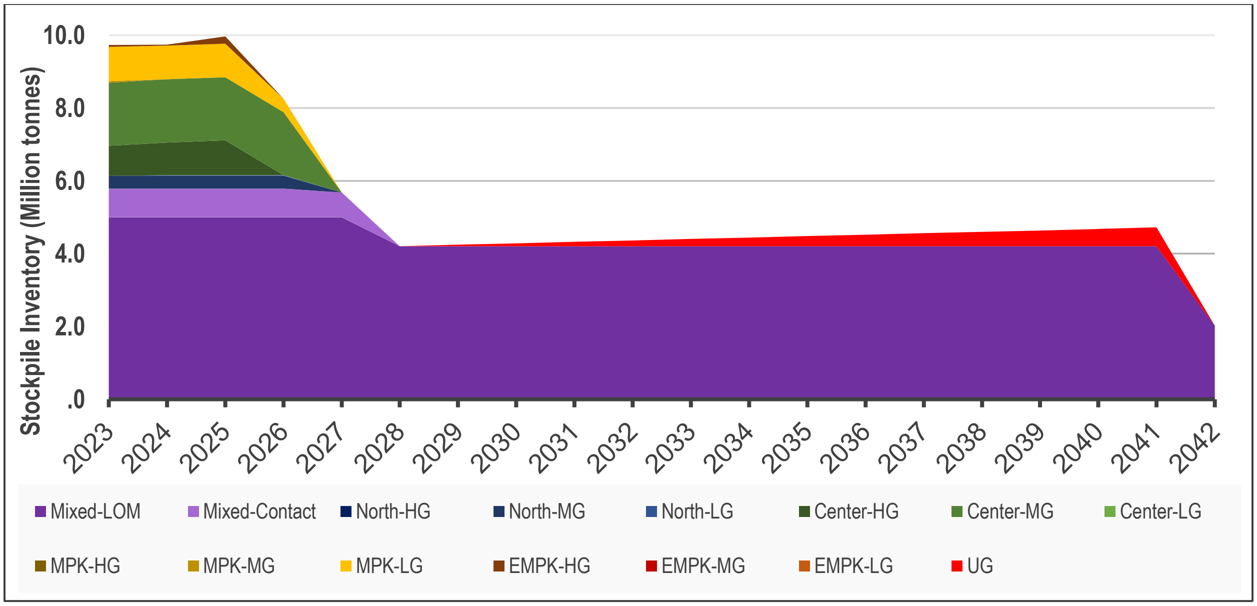
Figure 1-6: Summary of Mill Production



Source: JDS (2023)

A summary of the stockpile inventory opening balance is outlined in Figure 1-7.

Figure 1-7: Summary of Stockpile Inventory Opening Balance



Source: JDS (2023)

1.11 Recovery Methods

1.11.1 KDM Plant History

The KDM processing plant was designed by DRA Mineral Projects for operations beginning in 2012. It consisted of a milling, Dense Media Separation (DMS), recovery plant, associated crushing, screening and thickening systems. It was designed to process 2.5 Mt of run-of-mine (ROM) material per year with a single 200 t/h DMS module. The concentrate material from the DMS was subsequently treated through a 2.5 t/h wet X-ray recovery system for material reduction and diamond winning. This circuit was designed with adequate space to accommodate future expansions.

The KDM plant was upgraded in 2015 with the inclusion of XRT machines installed ahead of the DMS in order to recover large diamonds. This upgrade included the construction and commissioning of a new secondary (gyratory) crusher, tertiary crusher, upgrade to existing recovery building, XRT sizing and XRT diamond recovery circuits.

In 2017, the Mega Diamond Recovery Project was completed – which included adding XRT sorting technology ahead of the AG Mill. The objective of this project was to sterilize the feed of liberated diamonds above 50 mm by adding a recovery step up front.

In addition to the large-scale upgrades outlined, there have been several smaller improvements since 2017 including:

- Addition of a wet dust scrubber at the primary crushing section;
- Installation of a secondary gyratory crushing feed bin;
- Addition of wet dust scrubber at the pebble crushing section;
- Procurement of a mill relining machine;
- Incorporation of a Phase II audit XRT machine as part of the mainstream plant in a primary “scavenger” application / duty;
- Addition of a new XRT audit plant treating DMS, grits and XRT tails material;
- Restart of the dust suppression system:
  - The existing dust suppression system has been restarted at the end of August 2019 using Reverse Osmosis (R/O) plant filtered water quality to combat ore transfer point dust emissions.
- Expansion of the R/O plant capacity;
- Installation of new raw and process water tanks, complete with new pump manifolds and pumps;
- Decommissioning of recovery magnetic roll (or MagRoll) separators;
- Upgrade to the XRT sort house;
- XRT replacement / refurbishment;
- DMS/XRT floats (i.e., coarse ore stockpile):
  - Material from the coarse ore stockpile treated through the Bulk Sample Plant (BSP).
- Recovery plant red area tails dump treatment initiative regarding all associated stockpiles (inclusive of all tertiary crusher bypassed feed material).

1.12 Infrastructure

The UGP includes the use of existing and new infrastructure at the KDM, all designed to support operation of a 2.7 Mt/a UG mine and processing plant. Project construction over the past two years has led to the completion of most of the surface infrastructure components of the Project including but not limited to the following major items:

- 220/132 kV substation and 132 kV switchyard at Botswana Power Corporation's 400/220 kV Letlhakane substation;
- 29 km-long, 132 kV overhead powerline from the Botswana Power Corporation (BPC) Letlhakane substation to the KDM substation;
- 132/11 kV substation and switchyard located at the KDM minesite;
- Distribution of 11 kV power from KDM substation to the Project site;
- UGP pad surface substation and power distribution;
- Eight MW of diesel generator back-up power;
- Reverse-osmosis plant water supply plant;
- Sewage treatment plant upgrades;
- Phase 1 (two paddocks) of a new Fine Residue Deposits (FRD);
- 200-person capacity camp complex to support the construction workforce;
- Infrastructure pads and roadways;
- Surface sediment pond for managing UG dewatering;
- Buildings and facilities to support the operation including:
  - UGP office complex;
  - Change house for UG personnel;
  - Maintenance shops;
  - Warehouses;
  - Chemical grout mixing;
  - Lamp room;
  - Line out rooms;

- Training and meeting rooms; and
- Local first aid room.

- Shaft sinking infrastructure:
  - Shaft pre-sink winders and scotch derrick cranes (since removed);
  - Two shaft headframes and associated sub bank civil, steel, and pipework;
  - Three winder buildings equipped with four independently operated winders and control systems;
  - Shaft sinking ventilation fans, air coolers, and duct form rolling facility;
  - Shaft sub bank plenums for chilled air entry and mine service corridor;
  - Two dedicated concrete batch plants and aggregate storage facilities; and
  - Shaft mucking training tower.

The UGP will make use of existing operation infrastructure including the processing plant, site access road, airstrip, pit dewatering pipeline, maintenance facility, FRD (slimes storage facility), waste dump, coarse reject facility, explosive magazines and bulk fuel storage.

Major surface facilities remaining to be built for the UGP include the main mine exhaust fans, UG bulk air coolers, permanent P/S personnel and material winder, UG control room, and saline water management evaporators and containment pond.

Ongoing construction works include sinking of the P/S to 245 masl, V/S to 285 masl, ongoing pre-excavation grouting of Ntane hosted aquifers, construction of new TSF and expansion of UGP laydown infrastructure including workshops, laydowns, and office complexes.

1.12.1 Tailings Management

In response to evolving operational requirements and environmental considerations, Knight Piésold (KP) Consulting conducted a Feasibility Study in 2019 to enhance the design of the FRD 1. The technical report recommended raising the elevation of FRD 1 to 1,042 masl and constructing a new FRD 2 adjacent to it, with both phases reaching this final elevation. Subsequent design revisions in 2021 mandated height restrictions on FRD 1, limiting it to 1,031 masl, while FRD 2 was redesigned to accommodate tailings storage until the end of 2025 within the existing site boundaries. Commencing construction in 2022, FRD 2's final design includes two paddocks divided by a wall, utilizing a two-stage lifting process. Additionally, a site selection study in 2022 led to the identification of a new site for FRD 3 on the west of the existing facilities, with detailed design commencing in 2023. Both FRD 2 and FRD 3 adhere to a final elevation limit of 1,031 masl, aligning with the LOM tailings requirements. With deposition into FRD 2 underway as per planned OP production schedules, this report encompasses the feasibility designs for the Coarse Residue Deposit (CRD), FRD 2, and FRD 3, delineating a strategic framework for sustainable mine residue storage facilities.

1.13 Environment and Permitting

KDM has been operating since 2012, completed its latest Environmental Impact Assessment (EIA) / Environmental Management Plan (EMP) in 2020 (to incorporate the UGP) and received approval from the Botswana Department of Environmental Affairs during the same year.

A new EIA and its regulatory approval are still required for the proposed on-site storage and mechanical evaporation of significant volumes of produced saline groundwater (total dissolved solids (TDS) ±30,000 mg/l) in a lined pond between 2026 - 2030. By 2030, additional produced water disposal plans will need to be developed for the remaining LOM. This future plan is expected to be subject to an additional EIA and its regulatory approval.

The area hosting KDM, features farming and grazing activities. There are no artisanal mining activities at or near KDM. Lucara continues to enjoy very good relationships with local communities. Lucara updated its Stakeholder Engagement Plan, which includes a grievance mechanism, in 2023. Lucara's 2021 Human Rights Review identified access to water as a salient topic.

The area of the Mining License is covered by a mix of two vegetation types: mopane tree savanna and mopane shrub savanna, and due to grazing, farming and diamond mining, deemed to be modified habitat. The area features several species with conservation status. These include the White-backed vulture (*Gyps africanus*, critically endangered), African elephant (Loxodonta Africana, IUCN: endangered, but common in Botswana), as well as Devils claw (*Harpagophytum procumbens*) and Hoodia (*Hoodia currorii*), two plants which are included in Botswana's "Red Book".

Recent Archeological Impact Assessments (AIA) were carried out in 2018 and 2022. These did not reveal evidence of graves, cultural sites, archaeological sites, historical structures or buildings, within the area planned for development. The AIA reports' recommendations include archaeological monitoring during ground disturbing activities to deal with chance finds.

The Environment, Health, Safety & Community Relations Department comprises approximately 37 positions. The department includes dedicated health and safety, medical/wellness, sustainability, environmental, waste management, stakeholder engagement as well as corporate social investment line functions.

KDM received ISO 45001 certification for its occupational, health and safety system, and is pursuing ISO 14001 certification for its environmental management system. Lucara is also a certified Member of the Responsible Jewellery Council (expires March 2024, re-certification in progress), and is a Participant of the UNGC (latest 2022 Communication on Progress published in June 2022).

In line with its EIA/EMP, the mine continues to routinely monitor its environment and social performance using key performance indicators common to mining operations. Monitoring includes air quality, groundwater quality, water use, greenhouse gas emissions, waste management, biodiversity, environmental incidents, and community grievances. The results are reported to regulators, project financiers, and other stakeholders, including Lucara's third-party assured annual sustainability reports.

KDM is connected to the national grid and operates a diesel-fueled mobile fleet. In 2022, Lucara's greenhouse gas (GHG) emissions totalled 85,801 tonnes of carbon dioxide equivalent (tCO_{2e}) (Scope 1 and 2) and GHG intensity was 17.9 ((Total CO_{2e}(kt)/ore + waste rock mined (t)). Lucara continues to publicly disclose its annual GHG emissions, has developed a Decarbonization Strategy, commissioned a prefeasibility study for a large-scale solar PV project, and is exploring feasible options to significantly reduce its GHG emissions by 2030.

As part of the EMP, a Mine Closure and Rehabilitation Plan (MCRP) and associated costing was developed in 2018 and updated in 2020 and estimated to be $34 M for this report, including the UG. Lucara Botswana has provided financial guarantees totalling Botswana Pula (BWP) 240.0 million ($18.5 M) for reclamation obligations.

1.14 Operating and Capital Cost Estimates

1.14.1 Operating Cost Estimate

All cost figures quoted refer to US dollars (US$ or $) unless otherwise noted.

A summary of operating costs for the site is provided in Table 1-6. The operating costs below represent total LOM costs (including OP).

Table 1-6: Summary of Operating Cost Estimate

Operating Costs	Average Annual(1)	Life of Mine	Tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Open Pit Mining Costs	24.2	72.6	5.5	13.2	4
UG Mining Costs	29.5	413.2	37.0	11.2	24
Rehandle Costs	3.4	23.6	9.7	2.4	1
Process Costs	24.7	493.7	52.2	9.5	29
Other Power Costs	5.3	105.2	52.2	2.0	6
G&A	18.3	365.8	52.2	7.0	21
Cost of Sales	4.4	87.9	52.2	1.7	5
Corporate Charges (Botswana)	8.0	159.2	52.2	3.1	9
Total	**86.1**	**1,721.1**	**52.2**	**33.0**	**100**

Notes:
(1) Average cost per year in which costs occur.
(2) Tonnes processed in relation to operating cost.

Source: Lucara (2023) - Karowe FS Model V1.7

The mine operating cost estimate for KDM is based on a combination of experience, reference projects, first principle calculations, budgetary quotes, and factors as appropriate for an FS.

The main assumptions used to build up the operating costs are located in Table 1-7.

Table 1-7: Operating Cost Assumptions

Item	Units	Base	Source
Exchange Rates, Escalation, and Taxes			
South African Rand	ZAR:1USD	17.00	KDM
Botswana Pula	BWP:1USD	12.50	KDM
Escalation Rate	%	0	KDM
Value Added Tax (VAT)	%	14	BURS
Power			
Fixed Charge	BWP/month	92.78	BPC Line Power Delivered to site, excluding VAT
Demand Rate	BWP/kW	208.29	
Energy Charge	BWP/kWh	0.71	
Fuel			
Diesel Fuel, 50 ppm	BWP/L	15.06	Actuals 2023. Delivered to site, excluding VAT
Labour			
A2	BWP/month	130,787	KDM 2023 budgets, mid-range, fully burdened
B1	BWP/month	168,251	
B2	BWP/month	186,750	
B3	BWP/month	228,117	
B4	BWP/month	274,265	
C1	BWP/month	397,881	
C2	BWP/month	498,017	
C3	BWP/month	642,688	
C4	BWP/month	787,820	
D1	BWP/month	894,021	
D2	BWP/month	1,090,575	
D3	BWP/month	1,343,174	
D4	BWP/month	1,523,622	
E	BWP/month	1,817,693	

Source: JDS (2023) - LUCKAR14E - Cost Assumptions - RevA 2023.08.08

The total LOM operating costs for the UG operations are summarized in Table 1-8.

The operating cost estimate is based on an Owner's team workforce with year-round mining on two 12-hour shifts.

Table 1-8: UG Mining Operating Costs

Activity Operating Costs	Average Annual(1)	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Drill and Blast	9.3	65.2	1.76	16
Drawpoint Operations	3.8	52.9	1.43	13
Crush and Convey (UG)	0.7	10.5	0.28	3
Shaft Operations	4.0	55.4	1.50	13
Surface Haulage	5.0	69.9	1.89	17
Mine Maintenance	2.3	32.0	0.87	8
Mine General	6.4	89.7	2.43	22
Contingency	2.7	37.6	1.02	9
Total	**29.5**	**413.2**	**11.18**	**100**

Note:
(1) Tonnes processed are equal to tonnes mined less mine recovery.

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG_r2

A contingency has been included in the operating costs equal ten (10) percent of the sum of the direct operating costs to account for labour turnover, consumable growth, and unbudgeted work delays.

1.14.2 Capital Cost Estimate

The capital costs associated with developing and processing the material from the UGP are outlined below. LOM capital costs total $906 M, consisting of the following distinct phases:

- Pre-production capital costs total $683 M and are expended over an eight-year pre-production construction and commissioning period, of which three are already incurred; and
- Sustaining capital costs total $223 M which include stay in business costs for the current OP operation, incurred over the UGP period and costs incurred from commissioning of the UG until the end of the mine life.

Table 1-9 outlines the capital cost estimate.

Table 1-9: Summary of Capital Cost Estimate for LOM

WBS	Capital Costs	Pre-Production			Sustaining	LOM Total	Weighting
		Sunk	To Completion	Subtotal			
		(M$)	(M$)	(M$)	(M$)	(M$)	%
1000	Mining	140.4	253.1	393.5	124.8	518.2	63
2000	Site Development	12.7	13.4	26.1	6.6	32.7	4
3000	Process Plant	-	0.1	0.1	0.0	0.1	-
4000	Tailings and Mine Waste Management	-	-	-	42.8	42.8	5
5000	On-site Infrastructure	13.0	5.1	18.1	0.0	18.1	2
6000	Buildings and Facilities	2.1	3.1	5.2	0.0	5.2	1
7000	Off-site Infrastructure	23.3	0.4	23.7	0.0	23.7	3
8000	Project Indirects	9.4	21.7	31.1	1.4	32.5	4
9000	Owner Costs	63.6	89.9	153.5	0.0	153.5	19
Subtotal		**264.5**	**386.8**	**651.3**	**175.6**	**826.9**	**100**
10000	Contingency	0.0	31.9	31.9	13.3	45.2	
11000	Closure	0.0	0.0	0.0	34.0	34.0	
Total Capital Costs		**264.5**	**418.7**	**683.3**	**222.9**	**906.1**	

Notes:
*Numbers may not add due to rounding.

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

The details of the cost build up and main drivers of total costs are included below.

1.14.2.1 Mining

Mining capital costs include mining related surface infrastructure including shaft headframes, ventilation fans, cooling plants, and winder buildings. These costs are based primarily on actuals and contractor quotes and are largely already constructed. Shaft development costs are based on contractor quotes and are underway. UG development and infrastructure installations were built up from first principles using a mix of existing on-site contractor rates and expatriate contractors. Equipment and consumable costs are sourced locally where applicable. Table 1-10 provides a mining capital cost breakdown.

Table 1-10: Mining Capital Costs

WBS	Mining Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1100	Surface Infrastructure	35.5	26.3		61.8	12
1200	Shaft Sinking and Infrastructure	100.6	90.6		191.2	37
1300	UG Development		66.5	35.3	101.8	20
1400	UG Equipment	2.8	26.2	36.8	65.8	13
1500	UG Infrastructure	1.4	36.2	10.5	48.1	9
1600	Capitalized UG Operating Costs		7.4		7.4	1
1700	Infrastructure Sustaining			42.2	42.2	8
1000	**Total Mining**	**140.4**	**253.1**	**124.8**	**518.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.2 Site Development

Bulk earthworks were built up from first principles, based on existing contractor equipment and labour rates, or from contractor quotes. Outstanding site development costs are largely associated with permanent surface water management ponds and infrastructure.

Table 1-11: Site Development Capital Costs

WBS	Site Development Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
2100	Bulk Earthworks	10.4	0.6	-	11.0	34
2200	Site Roads	-	-	-	-	-
2300	Surface Water Management	0.1	6.2	6.6	13.0	40
2400	Dewatering	-	4.5	-	4.5	14
2500	Core Hole Drilling	2.1	2.1	-	4.3	13
2000	**Total Site Development**	**12.7**	**13.4**	**6.6**	**32.7**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.3 Process Plant

Minimal changes to the process plant have been identified as part of the FS. Sustaining capital costs include all stay in business costs to support the existing process plant and site infrastructure and have been budgeted by the existing site operations.

Table 1-12: Process Costs

WBS	Process Capital Costs	Pre-Production Sunk (M$)	Pre-Production Estimated (M$)	Sustaining (M$)	LOM Total (M$)	Weighting (%)
3100	Plant Upgrade	-	0.1	-	0.1	100
3000	**Total Process Plant**	**-**	**0.1**	**-**	**0.1**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.4 Residue Storage Facilities

The costs to expand the FRD facility to accommodate the additional slimes generated by the UGP have commenced and is partially complete. Future expansion costs were estimated based on engineered material take offs (MTOs) and existing contractor unit rates.

KDM does not plan for any capital projects at the Coarse Residue Facility nor the Waste Rock Storage Facility.

Table 1-13: Residue Storage Facility Costs

WBS	Residue Storage Facility Capital Costs	Pre-Production Sunk (M$)	Pre-Production Estimated (M$)	Sustaining (M$)	LOM Total (M$)	Weighting (%)
4100	FRD - Slimes	-	-	42.8	42.8	100
4200	FRD - Coarse	-	-	-	-	-
4300	Waste Rock Storage Facility	-	-	-	-	-
5000	**Total Tailings**	**-**	**-**	**42.8**	**42.8**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.5 On-site Infrastructure

On-site infrastructure capital costs include the supply and commissioning of the emergency backup power generator facility, surface power distribution infrastructure to the bulk air cooler, evaporation pond, and permanent winders, power factor correction equipment, surface water distribution lines, and Control Room building and infrastructure.

Table 1-14: On-site Infrastructure Costs

WBS	On-site Infrastructure Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
5100	Electrical Supply and Distribution	10.2	2.9	-	13.1	73
5200	Water Supply, Distribution, and Treatment	2.3	0.1	-	2.4	13
5300	Waste Collection and Treatment	0.5	-	-	0.5	3
5400	IT and Communications	-	2.1	-	2.1	12
5000	**Total On-site Infrastructure**	**13.0**	**5.1**	**-**	**18.1**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.6 Buildings and Facilities

Buildings and facility costs include remaining offices, ancillary buildings, change houses, and mine rescue center sustaining costs required to complete UG construction.

Table 1-15: Buildings and Facilities Costs

WBS	Buildings and Facilities Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
6100	Training Center	-	-	-	-	-
6200	Workshop and warehouse	0.2	0.1	-	0.2	4
6300	Mine Rescue Centre	1.4	1.3	-	2.7	51
6400	Offices	0.6	1.7	-	2.3	44
6500	Change house	-	-	-	-	-
6600	Access, Fencing, and Traffic Management	-	-	-	-	-

WBS	Buildings and Facilities Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
6000	**Total Buildings and Facilities**	**2.1**	**3.1**	**-**	**5.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.7 Off-site Infrastructure

Off-site infrastructure costs include all the direct construction costs associated with the construction of the new BPC electrical transmission line and associated substations, along with the costs associated with the construction of the contractor's camp.

Off-site development costs are largely complete with remaining budget allocated to close out and maintain the power transmission line and off-site accommodation facilities.

Table 1-16: Off-site Development Costs

WBS	Off-site Development Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
7100	Power Transmission Line	18.9	0.2	-	19.1	80
7200	Off-site Accommodations	4.5	0.2	-	4.7	20
7000	**Total Off-site Development**	**23.3**	**0.4**	**-**	**23.7**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.8 Project Indirect Costs

Project indirect costs cover camp catering, office rentals, bussing, and charter flights for personnel. Also included are freight and freight forwarding services, civil material testing, and waste rock haulage from the Project area to the waste rock dump.

Table 1-17: On-site Infrastructure Costs

WBS	On-site Infrastructure Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
8100	On-site Contract Services	4.0	17.4	-	21.4	66
8200	Temporary Facilities and Utilities	-	0.1	-	0.1	-
8300	Contractor Indirects	0.2	0.5	1.4	2.1	6
8400	Freight	4.7	2.4	-	7.1	22
8500	Temporary Accommodations and Expenses	0.5	1.4	-	1.9	6
8000	**Total Project Indirects**	**9.4**	**21.7**	**1.4**	**32.5**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.9 Pre-Production General and Administrative Costs (Owner's Costs)

Owner's costs are classified as the management, oversight and site operation costs that are incremental costs to develop the UGP. These costs are capitalized during the construction phase. Any Owner's costs that continue beyond the Project phase are then incorporated into the site G&A operating costs.

Owner's costs include:

- Engineering, Procurement, and Construction Management (EPCM) services;
- Owners' labour;
- 3rd party engineering services;
- Free issue materials including fuel, power, explosives, and cement;
- Project taxes and insurance;
- Human Resources;
- Pre-production operational charges; and
- Equipment fleet maintenance.

Table 1-18: Owner's Costs

WBS	Owners Capital Costs	Pre-Production Sunk (M$)	Pre-Production Estimated (M$)	Sustaining (M$)	LOM Total (M$)	Weighting (%)
9100	Pre-Production General and Administration	5.7	17.3	-	23.0	15
9200	Operational Charges	11.8	8.0	-	19.8	13
9300	Engineering, Procurement, and Construction Management	35.9	38.3	-	74.2	48
9400	Equipment Supply and Maintain	0.7	0.6	-	1.3	1
9500	Free Issue Materials	9.6	25.7	-	35.3	23
9600	Stay-In-Business Annual Budgets	-	-	-	-	-
9000	**Total Owners Costs**	**63.6**	**89.9**	**-**	**153.6**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.14.2.10 Closure

Lucara Botswana has provided financial guarantees totalling BWP 240.0 million for reclamation obligations, consisting of cash on deposit of BWP 40.0 million and a BWP 200 million standby letter of credit. Closure costs were originally prepared by Digby Wells in 2019 in preparation of the 2019 Feasibility Study and encompass the entire KDM site inclusive of the UGP. UGP closure costs have been estimated using a unit rate approach against the planned UGP infrastructure. Demolition and civil contractor quotes were used where possible for the original 2019 estimate and updated to 2023 rates by using a five-year historic escalation rate of 5.3% (World Data, 2023).

Table 1-19: Closure Costs

WBS	Closure Capital Cost	Pre-Production Sunk (M$)	Pre-Production Estimated (M$)	Sustaining (M$)	LOM Total (M$)	Weighting (%)
11110	Pit Buildings and Surface	-	-	4.2	4.2	12
11120	Open Pit and Dumps	-	-	13.0	13.0	38
11130	Slimes and Dams	-	-	8.7	8.7	25
11140	UG	-	-	2.3	2.3	7
11150	Monitoring	-	-	3.0	3.0	9
11160	Project Management	-	-	2.8	2.8	8

WBS	Closure Capital Cost	Pre-Production		Sustaining	LOM Total	Weighting
		Sunk	Estimated			
		(M$)	(M$)	(M$)	(M$)	(%)
11000	**Total Closure Costs**	**-**	**-**	**34.0**	**34.0**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

1.15 Economic Analysis

An economic model was developed to estimate the annual cash flows and sensitivities for KDM. All costs, diamond prices, and economic results are reported in US$ unless stated otherwise.

The main assumptions used in the economic model are:

- Discount rate of 8%;
- Nominal 2023 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- No escalation of costs or diamond price;
- No inflation;
- Canada corporate (Lucara Diamond Corp.) costs not included in the economic model results except as noted;
- Lucara Botswana corporate costs included in all economic results;
- Debt financing costs included;
- Working capital included; and
- The model excludes all sunk costs up to the base date of June 30, 2023 ($265M).

This technical report does not consider the UGP as a stand-alone asset nor evaluates it as a stand-alone economic cash flow. The cash flows presented herein are inclusive of the existing and ongoing OP operation which is near completion.

Table 1-20 through Table 1-22 show additional significant assumptions used in the 2023 FS.

Table 1-20: LOM Summary

Parameter	Unit	Value
Ore Processed	Mt	52.2
Mill Average Daily Production	kt/d	7.4
Mill Average Annual Production	Mt	2.7
Average Processing Grade	cpht	13.10
Diamonds Contained	k ct	6,834
Diamonds Recovered	k ct	6,834
Recovery*	%	100.0
Initial Capital Cost (inc. Contingency)	$M	418.7
Sustaining Capital Cost	$M	333.6
Life of Mine Capital	$M	752.3

*Processing recovery has already been factored in the resource estimate.

Source: Lucara (2023) - Karowe FS Model V1.7

Table 1-21: Economic Assumptions

Item	Unit	Value
Net Present Value (NPV) Discount Rate	%	8
Annual Escalation	%	0
BWP:US$ FX	BWP:US$	12.5
ZAR:US$ FX	ZAR:US$	17

Source: JDS (2023)

Table 1-22: Baseline Diamond Prices

Unit	Unit	FS
North	$/ct	273
Centre	$/ct	392
EM/PK(S)	$/ct	828
M/PK(S)	$/ct	707
Mixed Stockpile	$/ct	574

Source: JDS (2023)

Pre-tax estimates of Project values were prepared for comparative purposes, while post-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the post-tax results are only approximations.

The economic estimates in this technical report were generated from an engineering economic model appropriate for an FS-level report. The model should not be considered a cashflow model as defined by most international accounting standards but rather an indicative estimate of revenues and costs.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting revenues as part of this technical report. The mill head grades are based on sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, to obtain major equipment or skilled labour on a timely basis, or to achieve the assumed mine production rates at the assumed grades may cause actual results to differ materially from those presented in this economic analysis.

The reader is cautioned that the diamond prices and exchange rates, two of the biggest economic drivers, used in this technical report are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized in the future.

1.15.1 Results

The economic results for the Project, based on the assumptions outlined above are presented in Table 1-23.

Table 1-23: Economic Results - LOM Model

Parameter	Unit	After-Tax Results
$NPV_{8\%}$ including Canadian corporate costs	US$M	433.1
$NPV_{5\%}$ including Canadian corporate costs	US$M	562.5
$NPV_{8\%}$ excluding Canadian corporate costs	US$M	531.8
$NPV_{5\%}$ excluding Canadian corporate costs	US$M	684.5

Source: Lucara (2023) - Karowe FS Model V1.7

The LOM economic model does not calculate a meaningful Internal Rate of Return (IRR) as the UGP capital costs are partially offset by operating revenue during the years they are incurred.

The estimated total for the KDM undiscounted cashflow is $1,098M.

The post-tax break-even diamond price for the Project ($0 NPV @ 8% discount rate) is $483/ct or 65% of the assumed FS values.

1.15.2 Sensitivities

Sensitivity analyses were performed using diamond prices, UGP Capital Expenditure (CAPEX), and Operational Expenditure (OPEX) as variables. Mill head grade sensitivities mirror those of diamond value. The value of each variable was changed ±20% independently while all other variables were held constant. The Project is most sensitive to the carat price/head grade, followed by the OPEX and least sensitive to the CAPEX. The results of the sensitivity analyses are shown in Table 1-24.

Table 1-24: Sensitivity Results After-Tax (NPV @ 8%)

Variable	After-Tax $NPV_{8\%}$ (M$)				
	-20% Variance	-10% Variance	Base	+10% Variance	+20% Variance
Diamond Price	252.3	400.1	531.8	672.0	811.3
Mining Cost	556.8	544.3		519.2	506.7
Processing Cost	561.6	546.4		517.1	502.4
All Operating Costs	607.1	568.1		495.6	459.6
Upfront CAPEX	584.6	556.6		509.3	487.0
Sustaining CAPEX	548.1	539.9		523.6	515.5
All Capital Costs	602.3	565.4		501.2	473.1

Source: Lucara (2023) - Karowe FS Model V1.7

1.16 Project Development

The overall development period for the Project is estimated to be eight years from the start of detailed engineering to the UG reaching over 75% production capacity. To date, the UG site has been nearly fully developed with remaining infrastructure scheduled to be constructed as shaft sinking transitions into shaft equipping and lateral development.

The shafts are expected to be complete by H2 2026 with concurrent UG development commencing during shaft equipping. UG crushing and conveying infrastructure will commence in H2 2026, shortly followed by drawbell construction in H1 2027. Production stoping will ramp up through 2027, reaching full production in H1 2028. Additional details are provided in Table 1-25 below.

Table 1-25: UG Execution Schedule

	2023	2024		2025		2026		2027	
	H2	H1	H2	H1	H2	H1	H2	H1	H2
Production Shaft									
Sink	■	■	■	■					
Equip				■	■	■			
Ventilation Shaft									
Sink	■	■	■	■	■				
UG Construction				■	■	■			
Equip						■			
UG Development									
- Level: 245				■	■	■			
- Level: 285				■	■	■	■	■	■
- Level: 310					■	■	■	■	
- Level: 335						■	■		
- Level: 340							■	■	
- Level: 380							■	■	
- Level: 470								■	■
- Level: 580								■	■
- Level: 670									
UG Infrastructure									
Pump Station							■	■	
Workshop								■	■
Crusher							■	■	■
Conveyor							■	■	
Draw bells								■	■
UG Production									
- Level: 380									■
- Level: 470									■
- Level: 580									
- Level: 670									

Source: JDS (2023)

1.17 Conclusions

It is the conclusion of the QPs that this technical report contains adequate data and information to support an FS-level report. Standard industry practices, equipment and design methods were used in the FS. Since the 2019 FS, the UGP has advanced considerably in terms of financing, detailed engineering and construction while the OP mine and processing facility have operated well and maintained targeted production.

Most of the surface infrastructure relating to the UGP is now established and the main focus going forward for the UG is completion of detailed engineering, selection of a lateral development contractor, continued groundwater control and continued focus on meeting and improving the Project schedule and budget.

The most significant potential internal (controllable) risks associated with the Project are; uncontrolled stope back failure, uncontrolled dilution, operating and capital cost escalation, schedule delay, the ability to dewater and depressurize the mine (both OP and UG) ahead of production, ability to grout and manage water inflows during pre-production, ability to manage gas from the kimberlite and host rock structures, accuracy of the Mineral Resource Estimate, skilled contractor and employee personnel availability (with corresponding work permits for expatriates). A more complete risk table and mitigation initiatives matrix is included in the body of this report.

To date, the QPs are not aware of any fatal flaws for the UGP.

1.18 Recommendations

Some of the main recommended work is summarized below and all costs are part of the construction and operating costs within this technical report, the following work is recommended:

- Continued work on verifying rock stresses and rock mass behaviour;
- Careful draw control during stoping and continued monitoring of stope back conditions;
- Monitoring and re-modelling of groundwater pressures, dewatering achievements and water inflow conditions;
- Monitoring of large diamond distributions and recoveries;
- Further investigation into south lobe shape and internal, localized kimberlite domain boundaries;
- Continued reconciliation of the Mineral Resource model; and
- Continued exploration of GHG reduction opportunities; and Optimization of tailings management.

2 INTRODUCTION

This Technical Report was compiled by JDS Energy & Mining Inc. (JDS) with the assistance of other consulting companies listed in Section 2.1.1.

JDS Energy & Mining Inc. (JDS) was commissioned by Lucara Diamond Corp. (Lucara) to lead an updated Feasibility Study (FS) for the Karowe Diamond Mine (KDM) UG Mine Expansion Project (UGP or Project) currently being built to establish UG (UG) mining after the completion of OP mining. This technical report describes the combined life of mine (LOM) OP and UGP as well as highlight the contribution of the UG to the overall plan economics.

This report is updated from the original 2019 UGP FS and includes the following major changes:

- Advancement of detailed engineering designs;
- Re-modelling of the hydrogeological conditions;
- Modifications to the mine design;
- Modifications to the mine, mill and project construction schedules;
- Re-estimation of the current operations budgets and Project capital and operating cost projections;
- Change to UG dewatering and grouting methodology;
- Changes to groundwater management on surface;
- Consideration of Project construction progress (infrastructure and UG development) to the effective date of this report;
- Revised economic modelling with updated diamond prices and exchange rates, exclusion of sunk costs and inclusion of financing costs; and
- Revised waste management plans.

This report was prepared using guidance from the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101.

The Mineral Resource and Reserve estimates reported herein were prepared using guidance from the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, November 29, 2019 and "Rock Hosted Diamond Guidance", March 1, 2008.

This report has an effective date of June 30, 2023.

2.1 Qualified Persons and Responsibilities

The results of this FS are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Lucara and the QPs. The QPs, with the exception of John Armstrong who is a Lucara employee, are being paid a fee for their work in accordance with normal professional consulting practice.

2.1.1 Scope of Work

This technical report summarizes the work of several consultants with the scope of work for each company listed below, which combined, comprises the total Project scope.

- **DRA Botswana (Pty) Ltd.**: Mineral processing description;
- **Itasca Denver**: Hydrogeological modelling;
- **JDS Energy & Mining Inc.:** Mine engineering, production planning, cost estimation, economic modelling, report compilation;
- **K-Met Consultants Inc.**: Metallurgical testing;
- **Knight Piésold (Pty) Ltd. (Botswana) (KP)**: Waste material management, tailings management facility, geotechnical investigations, coarse residue deposit, FRD, stability assessment, stormwater management, water balance;
- **PRIZMA LLC**: Environment, permitting and social considerations;
- **SRK (South Africa)**: Geotechnical analysis; and
- **SRK Consulting (Canada) Inc.**: Geology, Mineral Resource estimation and UG material flow simulation.

2.1.2 Qualifications and Responsibilities

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions / associations. All QPs are independent except John Armstrong, Lucara's Vice President Technical Services. The QP scopes of work, responsibilities and their specific report sections are shown in Table 2-1.

Table 2-1: QP Responsibilities

QP	Company	QP Responsibility / Role	Report Section(s)
John Armstrong, Ph.D., P.Geo.	Lucara Diamond Corp.	History, Deposit Types, Exploration, Drilling and Sample Preparation, Analyses and Security, Size Frequency and Value Models, Market Studies	6, 8, 9, 10.1, 10.2, 11, 14.4, 19
Brandon Chambers, P.Eng.	JDS Energy & Mining Inc.	Mineral Reserve Estimate	15
Gord Doerksen, FEC, P.Eng.	JDS Energy & Mining Inc.	Overall Project Management, Infrastructure and Economics	1 to 5, 12 (except 12.1 and 12.2), 13.1, 13.2, 13.4, 16.6.1, 17.4.9, 18, 20.5, 22.3 to 22.5, 23 to 29 except 27. 2
William Joughin, Pr. Eng, FSAIMM, FSANIRE	SRK (South Africa)	UG Geotechnical Considerations	16.3
Houmao Liu, Ph.D., PE	Itasca Denver	Hydrogeological Considerations and Water Management	16.4, 27. 2
Kelly McLeod, P.Eng.	K-Met Consultants Inc.	Comminution	13.3
Matt Moss, P.Eng.	JDS Energy & Mining Inc.	UG Mining	16 (except 16.3, 16.4, 16.6.1), 21, 22 (except 22.3 - 22.5)
Mehrdad Nazari, MBA, MSc	PRIZMA LLC	Social, Environment and Permitting	20 (except 20.5)
Cliff Revering, P.Eng.	SRK Consulting (Canada) Inc.	Mineral Resource Estimate	12.2, 14 (except 14.4)
Justin Teixeira, Pr. Eng.	Knight Piésold	Tailings Engineering	18.8
Lehman van Niekerk, Pr. Eng.	DRA Projects	Mineral Processing	17 (except 17.4.9)
Kimberley Webb, P.Geo.	SRK Consulting (Canada) Inc.	Geology	7, 10.3, 12.1

2.2 Qualified Person Site Visits

In accordance with National Instrument 43-101 guidelines, all QPs, except for Kelly McLeod have visited KDM as per Table 2-2. 2023 site visits by QPs Revering, Webb and van Niekerk were not undertaken as no new work was done in the processing plant and resource drilling since their last visit as confirmed by QP Doerksen. QP Justin Teixeira relied on site visit communication with Knight Piésold engineers Amos Ditsela and Saumil Parmar.

Table 2-2: QP Site Visits

Qualified Person	KDM Visit Dates	Description of Inspection
John Armstrong	Regular visits since 2013	Full operation reviews and inspections of plant, mine and project work.
Brandon Chambers	Aug 28 – Sep 6, 2019 Regular visits since 2022	Full update review and inspection of the UGP and the OP mine. Regular meetings and discussions with various technical and management personnel, inclusive of the on -site OP engineering team. On-site engineering and construction oversite of the UGP.
Gord Doerksen	Approximately quarterly visits since 2019	Full update review and inspection of the UGP and discussions with various technical and management personnel.
William Joughin	Nov 2022	View the country rock and kimberlite exposures in the OP. View country rock exposures in the shafts. Examine core from shaft core holes.
Houmao Liu	April 2022	Visited the OP and dewatering operations. Visited the tailing facilities. Examined cores. Met with hydrogeologist, mine planning, and Geotech teams at the mine regarding depressurization of pit slope.
Kelly McLeod	No minesite visit	Visited the metallurgical lab during comminution testing.
Matt Moss	Regular visits (at least quarterly) since 2020	On-site engineering and construction oversite of the UGP. Regular meetings with site Mining Team, Geologist, Geotechnical, Hydrogeological Engineers, and sub-contractors. Visits to the OP and primary crushing plant, core sheds, magazine, and other site infrastructure. Visits to diamond sales office in Gaborone.
Mehrdad Nazari	Apr 27-28, 2021 Feb 14-25, 2022 Feb 13-17, 2023	Engagement with site staff and stakeholders to verify EIA, SIA and EMP findings. Examination of site conditions. Examination of consultant procedures to generate monitoring data and findings.
Cliff Revering	May 14-17, 2019	Review of mine geology, production tracking, mine reconciliation, process plant, geology core shacks and drill core. Discussions with various technical and management personnel. Review of Lucara's Diamond Sales and Marketing Office in Gaborone, Botswana. Inspection of run-of-mine diamond parcel from early May 2019.
Justin Teixeira	Dec 12, 2018 Sep 2-3 2019	Project scope, Slimes and tailings operation review, information gathering from various technical/plant personnel.
Lehman van Niekerk	Sep 2-3, 2019	Review of the surface treatment plant process and discussions with various technical and management personnel.
Kimberley Webb	June 14-22, 2017 Jun 11-15, 2018 May 8-17, 2019	Design kimberlite core logging procedure and train geologists. Review of OP exposures, kimberlite drill core from 2017 drilling and from 2018-2019 FS program and geological sampling protocols. Review of Lucara's Diamond Sales and Marketing Office in Gaborone.

2.3 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial and other units that are commonly used in industry (e.g., carats for diamonds). A carat is a unit of mass equal to 200 milligrams.

All currency figures quoted in this report refer to United States (US) dollars (US$, USD or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, JDS does not consider them to be material.

An appropriate number of significant figures has been used to reflect the order of accuracy and the degree of precision of the available numerical data.

2.4 Sources of Information

This report is based on information collected by the QPs during site visits and work conducted on the KDM site in the past four years since the last FS including but not limited to information provided by Lucara and other project specialists. Other information was obtained from the public domain. Discussions and data acquisition with Lucara personnel included:

- Lucara actual operating performance and data acquired through ongoing operations;
- Lucara planned budgets, schedules and initiatives;
- Inspection of KDM and UGP including processing facility, waste facilities, OP mine, infrastructure, shafts and drill core;
- Review of drilling data collected by SRK and others as part of the 2019 FS field program and 2021 shaft center-line core holes;
- Regional and international vendors;
- Past internal and external reports including the 2019 FS;
- Independent laboratory tests and analyses;
- Economic model structure and input review and discussions; and
- Additional information from public domain sources.

The QPs have no reason to doubt the reliability of the information provided by Lucara and others and the information has been verified by the respective QPs.

2.5 Terms of Reference

The terms of reference for the detailed design of the KDM tailings storage facilities encompass the planning and engineering required to develop a safe, environmentally sustainable, and efficient storage system for mineral processing waste. The purpose of this detailed design is to create a facility that adheres to industry best practices, regulatory guidelines, and environmental standards, ensuring the containment and management of tailings in a manner that minimizes potential risks to both human health and the surrounding ecosystem. The designs have considered factors such as topography, geotechnical characteristics, climate, and operational requirements, with a focus on constructing a facility that facilitates effective tailings deposition, water management, and long-term stability. Additionally, the design prioritizes monitoring systems and emergency response plans to address any unforeseen issues and contribute to the overall safety and sustainability of the mining operation.

3 RELIANCE ON OTHER EXPERTS

The QPs' opinions contained herein are based on; the QP's own work, information provided by Lucara and numerous internal and external contributors throughout the course of this technical report. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

The QPs used their experience and knowledge to determine if the information from previous reports was suitable for inclusion in this Technical Report and have adjusted information that required amending.

For this FS, JDS utilized an economic model developed by Lucara Diamond Corp. Lucara provided inputs to the economic model including G&A costs, OP mining costs, sustaining capital costs outside of the UGP, financing costs and details, sunk costs and tax and royalty payment calculations. Gord Doerksen reviewed and takes responsibility for the economic model and all Lucara's model inputs.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Botswana is a land-locked sub-Saharan country situated north of South Africa, east of Namibia and west of Zimbabwe. KDM sits on the eastern edge of the Kalahari Desert and south of the Makgadikgadi Pans and is about 15 km south-west of the Village of Letlhakane. The geographic coordinates of KDM are 25° 28’ 13” E / 21° 30’ 35” S or by UTM coordinates: 341,590 m East and 7,621,640 m South.

The Republic of Botswana is a parliamentary multi-party democracy that peacefully achieved independence from Great Britain in 1966. Elections are held every five years and although there are normally numerous parties competing, the country has been ruled by the Botswana Democratic Party since independence. Botswana is serviced by well-established governing institutions in the form of various ministries and agencies.

Botswana is one of the richest sub-Saharan Africa countries and is consistently rated as having one of the lowest perceived corruption levels in the region. It is one of the world’s largest diamond producers by value, driven mainly by the massive Jwaneng and Orapa Mines owned by the Debswana Diamond Company an equal partnership of the De Beers Group and the Government of the Republic of Botswana. Mining is governed by the *Mines and Mineral Act* and this act 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.

Figure 4-1: Project Location Map

Source: GeoAtlas (2018)

4.1.1 Types of Mineral License in Botswana

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

- Prospecting License: A prospecting license is valid for an initial period of up to three years with two renewals each not exceeding two years each. At the end of each period, the prospecting area is reduced by half or at lower proportions 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 program;

- Retention License: This license provides for prospectors who deem a project economically unviable in the short-term. The first three-year license remains exclusive while a second three-year license provides limited rights for third parties to reassess a prospect;

- Mining License: This license is initially valid for a period of up to 25 years, as is reasonably required to carry out the mining program. The holder of a license 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; and

- 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 kilometer. It is initially issued for five years, with unlimited renewal periods of up to five years each.

Figure 4-2: OP and UG Pad Site Layout

Source: Lucara (2023)

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 23.3);
- 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; and
- There is 15% taxation on revenues for downstream cutting and polishing of diamonds.

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

The Property is governed by Mining License (ML) 2008/6L, issued in terms of the *Mines and Minerals Act* 1999, Part VI, and covering 1,523.0634 ha.

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

ML 2008/6L is 100% held by Boteti, a company incorporated in Botswana. The ML was originally issued on October 28, 2008 and was updated on May 9, 2011 to increase the area to the current extent. The license was renewed in 2021 for a period of 25 years and expires on January 03, 2046. The Government of Botswana holds no equity in the Project. The corner points and geographic location are shown in Table 4-1, Figure 4-1 and Figure 4-2.

Table 4-1: List of Corner Points of ML 2008/6L

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

Source: Nowicki et al. (2018)

Figure 4-3 is an aerial photograph of KDM and has been marked up to highlight the OP, the stockpiles, waste dumps, fine tailings dam and coarse tailings storage facility. The process plant is located to the east of the OP.

Figure 4-3: Aerial View of the Mine Site

Source: Lucara 2023

Figure 4-4: Areal View of the KDM Permit Area

Source: Lucara 2023

4.3 Permitting Rights and Agreements Relating to KDM

4.3.1 Surface Rights

The surface area of ML 2008/6L 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 ML 2008/6L surface area and the access road. These rights will remain in force until 2046.

4.3.2 Taxes and Royalties

KDM is taxed according to a prescribed schedule of the Income *Tax Act*. Profits from KDM 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 authorized to offset withholding taxes against the variable income tax liability.

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

4.3.3 Obligations

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

- Commence production on or before the date referred to in the program of mining operations as the date by which he intends to work for profit;
- Develop and mine the mineral covered by his mining license in accordance with the program 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; and
- Furnish the Ministry with a copy of the annual audited financial statements within six months of the end of each financial year.

Lucara Botswana has met all of these obligations.

4.3.4 Environmental Liabilities

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

4.3.5 Permits

A list of permits held or in the process of being acquired by KDM is presented in Table 4-2 and discussed in detail in Section 20.

Table 4-2: KDM Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIA Permit	DEA/BOD/CEN/EXT/MNE 015(7)		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 license	Dept. of Water Affairs	*Water Act*
Borehole Certificates	In Place	Valid for the duration of the mining license	Dept. of Water Affairs	*Boreholes Act*
Dumps Classification	All clarified	All dumps active	Dept. of Mines	*Mines, Quarries, Works and Machinery Act*
Surface Rights	LT/SLB/B/1 IV (231)	Valid for the duration of the mining license	Ngwato Land Board	*Tribal Land Act*
Radiation License	BW0315/2021	6-Nov-25	Radiation Inspectorate	*Radiation Protection Act*
Incinerator Permit	DJM 2020/08-05	31-Aug-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Waste Water Treatment Plant	WMF01/2022/11/20-WWTW/Karowe Diamond Mine	30-Nov-24	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Landfill	WMD/22-2022/304-10/LF/Letlhakane	31-Dec-24	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Salvage yard	WMF/20-2022/20-11//Letlhakane	31-Dec-24	Department of Waste Management Pollution Control	*Waste Management Act*
Permit to purchase, acquire and Possess Explosives	F001/2022	31-Dec-24	Dept. of Mines	*Explosives Act*
Permit to carry bulk explosives	EX.10-07/2023 Vehicle No: B868BOY	31-Dec-24	Dept. of Mines	*Explosives Act*
Explosives magazine license	00003513A	31-Dec-24	DME	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	00003512A	31-Dec-24	DME	*Explosives Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Permit to import and possess explosives	Jan-22	31-Dec-24	DME	*Explosives Act*
Application for restricted blasting license		N/A	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B681BMU	23-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B693BRO	24-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B339BPM	25-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B429BJB	26-Jun-2024	DOM	*Explosives Act*
License to manufacture explosives	E-PCE0410/2022 Vehicle No: B693BRO	31-Dec-2024	DME	*Explosives Act*
Box storage for conveyance and Storage of explosives	F01/22 F02/22 F03/22 F04/22	31-Dec-2024	Dept. of Mines	*Explosives Act*
Blasting License for magazine master	In Place	valid and appointment renewed annually	Dept. of Mines	*Explosives Act*
Airstrip License	B509	LICENCE NO. B509	Civil Aviation	*Aviation Act*
Generator Licenses		Once off	BERA	*BERA Act*
Solar photovoltaic plant		Once off	BERA	*BERA Act*
Standby Generator Licenses		Once off	BERA	*BERA Act*
Mining License	2008/L6	March-46	Dept. of Mines	*Mines & Minerals Act*
License to possess and use radioactive sources	BW061/2022	1-Aug-24	Radiation Protection Inspectorate	*Radiation Protection Act* (No. 22 of 2022)

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Winder Engine drivers	M35 M 1 (20)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Kibble Winder 10 - 039 - Ventilation shaft	M35 M 1 (30)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Kibble Winder 10 - 069 - Production shaft	M35 M 1	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Kibble Winder 10 - 071- Ventilation shaft	M35 M 1 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Vertical Shaft Mucker (VSM)	M35 M 1 (33)	15-Oct-24	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Vertical Shaft Mucker (VSM)	M35 M 1 (14)	15-Oct-24	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Approval letter for charging units	11-May-00	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO2/22	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO3/23	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO4/24	N/A	Dept. of Mines	*Explosives Act*
Mobile rescue winder - truck mounted	M35M (16)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Capacity increase for magazine No. 385	EX.5 XXII (27)	N/A	Dept. of Mines	*Explosives Act*
Drill Approval Sandvick boom drill rig	2 C 66 XXV11	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Kibble Winder 10 - 069 - Production shaft	DOM 6/13/51(8)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Kibble Winder 10 - 069 - Ventilation shaft	DOM 6/13/51(9)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Permit to purchase, acquire and Possess Explosives	E - PPAP0035/2024	31-Dec-24	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	E-PCE0161/2024	30-Jun-24	Dept. of Mines	*Explosives Act*

Source: Lucara (2023)

4.4 Property Risks

The QP is not aware of any significant or anomalous factors or risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

KDM is accessed by 15 km of well-maintained all-weather gravel road from the tarred A14 Highway linking Serowe to Orapa road at the edge of the village of Letlhakane. There are good telecommunications including cellular telephone networks in the area. Letlhakane is reached from the major cities of Gaborone, Maun and Francistown by good quality tarred roads that also extend into neighbouring countries.

There is a 1,500 m all-weather gravel airstrip on the KDM property. There are no scheduled flights into KDM, but charter flights are scheduled at times. International charter flights, normally from South Africa, must stop and clear customs and immigration at an international airport, normally the capital, Gaborone, prior to landing at KDM.

The closest airport with commercial flights is Francistown, approximately 200 km to the east and two and a half hours away by road. The Maun airport is located 350 km to the northwest and is very active with commercial flights from South Africa mainly supported by the tourism industry of the Okavango Delta. Driving time between Maun and Letlhakane is about 4 hours. There is also a private airstrip within the nearby Debswana-controlled Orapa Township.

5.2 Climate

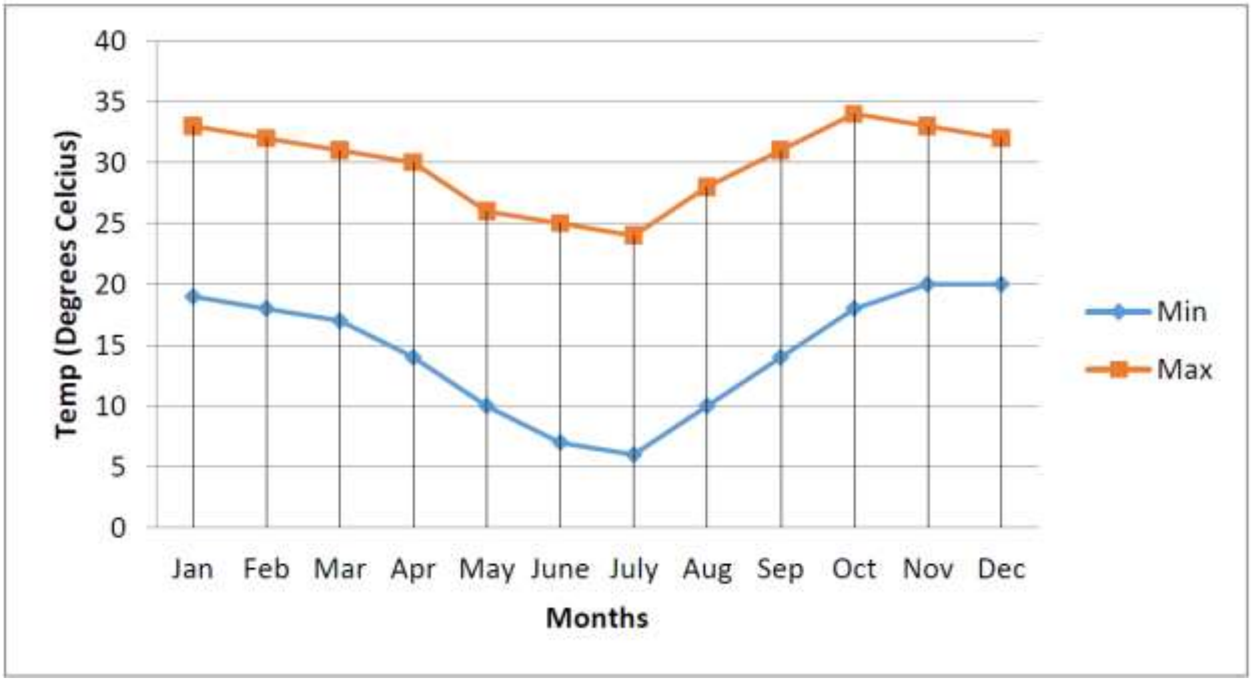
The climate in the Project area is hot and semi-arid, with an average annual rainfall of about 400 mm falling between November and March. Rainfall often occurs in short-duration, intense downpours from discreet cells often associated with lightening and is heaviest in January, February and March.

October to April is also the hottest time of year with daytime highs above 30°C, and extreme days up to 40°C, with lows around 20°C. Daytime high temperatures in the cooler dry season average about 25°C with nighttime lows below 10°C During the coolest months.

Winds in the Letlhakane area are predominantly easterly blowing at different speeds at about 20% of the time. Winds can become strong and gusty at times, especially during the months of August and September. During this time there may be considerable amount of dust in the air and visibility may be reduced. Calm conditions are experienced 17% of the time based on recent surveys.

The local climate does not impede construction or operating work although dirt roads may become difficult to travel on for short periods (a few hours) during or immediately after the heaviest downpours. A summary of monthly average temperatures is shown in Table 5-1.

Table 5-1: Letlhakane Monthly Temperature Averages



Source: Karowe Diamond Mine Climatic Data (2018)

5.3 Physiography

The Property is at an elevation of 1,022 masl. The region is very flat with no hills or significant topographical anomalies. The general ground surface 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 mine 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 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.

Large herbivores found in the region include gemsbok, hartebeest, wildebeest, kudu, ostrich, springbok, duiker, impala, warthog and steenbok. Wildlife in the immediate vicinity of KDM is scarce, mainly due to increased human presence, related livestock farming activities and mining activities although some of the more common species (warthog, steenbok, impala and kudu) are occasionally seen locally. Elephants have also been sited at KDM.

Species of small herbivores recorded in the region are vervet monkey, cape hare, scrub hare, porcupine, spring hare, tree squirrel and damara mole rat.

Carnivorous animals or predators recorded but rarely reported near the Project area include leopard, lion, brown hyena, spotted hyena, black-backed jackal and mongoose.

Some various types of snakes commonly occur in the area, and these include puff adder, boomslag, twig snake, black mamba, Mozambique spitting cobra and python. Tortoises can be abundant, especially on the sandier areas where the grass and bush cover is abundant.

Bird life in the Project area is not particularly rich but includes doves, sparrows, drongos, starlings, hornbills, weavers, vultures and egrets.

5.4 Local Resources and Infrastructure

Letlhakane village is the closest settlement and offers basic facilities. In 2011, the census noted that Letlhakane had a population of 23,000 but is likely to be closer to 30,000 presently.

Letlhakane has eight public schools which are four primary schools, three junior secondary and one senior secondary school.

Basic goods and services including fuel, clothing, groceries, banks, restaurants, accommodations, hardware, medical care, etc. are available in the village with varying degrees of selection. Cellphone coverage is generally good.

Industrial services such and heavy equipment rentals, parts suppliers and fabrication shops are very limited and tend to be sourced from Gaborone or South Africa.

Diamond mining in the area surrounding Letlhakane started in 1971 when operations commenced at the nearby Orapa Mine, one of the largest diamond mines in the world. There is some qualified and experienced mining-related manpower in the immediate vicinity, but most surplus skilled labour is found in past-producing mining areas like Francistown and Selebi-Phikwe. The Gaborone area is also a significant source for skilled people.

The Government supplies electrical power on commercial terms to KDM through the Botswana Power Corporation’s (BPC) national grid. For the Project the BPC substation at Letlhakane was expanded and a new 132 KV powerline was run between the Letlhakane substation and a new KDM substation. The two new substations and power line were funded and constructed by the Project and then ownership turned over to BPC to operate.

Water for KDM and the Project is derived from a strong aquifer at the contact of the Ntane Sandstone Formation and the overlying Karoo basalt. Currently, KDM has surplus water and transfers the excess to the Orapa Mine. Water is also recycled within the KDM facility from the tailings area to the plant.

A fully equipped 200-person construction camp was built for the Project and is located near the main gate within the mine permit area.

Surface rights have been secured over the Mining License and provide sufficient space for rock dumps, water management facilities, tailings dams and mine infrastructure.

The KDM property within its surface rights area has the necessary room and existing infrastructure or planned infrastructure to conduct the LOM mine plan as per this report including but not limited to; mining operations, electrical power supply, water supply, mining personnel, tailings storage areas, waste disposal areas, and processing plant facilities.

6 HISTORY

The contents of Section 6 are extracted from Nowicki et al. (2018) and 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 program that between 1967 and 1970 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 September 16, 1972 until December 15, 1975. Under the grant, De Beers carried out evaluation and the delineation of kimberlites discovered previously. In addition, they carried out reconnaissance and detailed soil sampling.

Little data from the initial discovery and evaluation of the AK6 kimberlite is available, but it is known that the discovery was made from the interpretation of an aeromagnetic survey. The kimberlite was delineated with 44 percussion boreholes, 20 of which were recorded as intersecting kimberlite and 24 as intersecting basalt. De Beers interpreted the AK6 kimberlite to have an area of 3.3 ha. A series of three 20 foot (~6.5 m) deep pits excavated in 1973 gave a grade of 0.07 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 programs 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. PL 17/86

The current AK6 kimberlite and Karowe Mine lies within former prospecting license PL 17/86 held by Debswana from July 1, 1986 until January 24, 1998. The kimberlite lies within the area dropped at the second relinquishment stage. The primary focus of the work programs on the license was 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 was 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. PL 1/97

PL 1/97 was issued to De Beers Prospecting Botswana (Pty) Ltd. (Debot) on February 1, 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. PL 13/2000

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 license was renewed on March 31, 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 comprised of basalt breccia.

In December 2003, De Beers started a program 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 license had been included in the Boteti Joint Venture.

6.5 The Boteti Joint Venture

On April 17, 2004, a joint venture agreement was entered into between Kukama Mining and Exploration (Pty) Ltd. and Debot for seven prospecting licenses in the Orapa area totalling 1,344.27 km^2, including 29 previously discovered kimberlites. This included PL 13/2000 and AK6. A twelve-month work program was carried out per the heads of agreement, which resulted in the signing of a formal joint venture agreement on October 20, 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 11.

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

On May 24, 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 $49 M. Government approval which, under the *Mines and Minerals Act* Section 50 was a condition precedent for this transaction, was given on December 18, 2009. In April 2010, African Diamonds exercised its option to increase its interest by 10.268% at a cost of $7.3 M. 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 December 20, 2010, Lucara secured a 100% interest in the AK6 Project pursuant to an arrangement which combined Lucara with African Diamonds Limited under a British court-approved scheme of arrangement.

On July 25, 2011, Lucara commenced trading its shares on the Botswana Stock Exchange, and on August 29, Lucara commenced trading its shares on the TSX main exchange (after moving from the TSX Venture Exchange). On November 25, 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. The first production diamonds were recovered in April of 2012. 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 $4.5 M, and in September 2019, recovered a 9.7 ct Type II blue diamond along with a 4.1 ct gem quality pink diamond. Karowe has established itself as a producer of large gem quality Type II white diamonds as well as a producer of rare gem-quality, coloured diamonds.

In 2015, the plant optimization project at the Karowe Mine was completed, with the objective being to modify the process plant to treat harder, more dense material at depth and improve the recovery of large + 35 mm diamonds. The plant upgrade introduced XRT bulk sorting to the flow sheet to for overall process improvement and recovery of large diamonds. In November of 2015 the Karowe Mine recovered the 1,109 ct gem quality Lesedi la Rona (sold for $53 M) and the 813 ct Constellation diamond (sold for $63 M).

During 2017, a drilling program was initiated at the Karowe Mine to test the AK6 kimberlite at depths below 400 m. Mineral Services Canada was contracted to assist in the development of the sampling program and internal geology updates that allowed for an updated resource estimate for the inferred portion of the Karowe Mine resource estimate, between a depth of 400 to 600 m below surface (600 to 400 masl). This study was completed mid-2018.

In September 2017, Lucara announced the completion of two diamond recovery capital projects: The Mega Diamond Recovery (MDR) project and Sub-middles XRT project. The commissioning of the MDR and Sub-middles circuits advanced Lucara’s ability to recover diamonds prior to the comminution process where diamond damage may occur and thus maximize value for its exceptional diamonds. The Sub-middles circuit allows for diamond recovery down to 4 mm through XRT sensor-based sorting without DMS concentration.

In November 2017, Lucara announced the results of its Preliminary Economic Assessment (PEA) for UG development at the Karowe Mine (the Karowe UG PEA). In Q3 2018, it was determined that the updated 2018 resource estimate, in conjunction with geotechnical and hydrogeological field programs already underway in 2018 were sufficiently detailed to support conversion of the planned pre-feasibility study into a feasibility study.

In April 2019, Karowe recovered the 1,758 ct Sewelo diamond, the largest diamond recovered at Karowe and from Botswana.

In November 2019, Lucara announced the positive results of an FS for an underground mine at Karowe.

In the first quarter of 2020, due to travel restrictions imposed to reduce the spread of COVID-19, Lucara received approval from the Government of the Republic of Botswana (GRB) to temporarily move quarterly tender sales to Antwerp, Belgium from Gaborone, Botswana. Mining was declared na essential service by the GRB and Karowe Mine continued to operate throughout the COVID-19 period with appropriate measures in place to maintain operations.

In July 2020, Lucara entered into a sales agreement with HB Antwerp for all stones greater than 10.8 ct in size. Under this agreement, +10.8 ct stone production from the Karowe Mine are sold at prices based on the estimated polished outcome of each diamond, determined through state of the art scanning and planning technology, with a true up amount payable to Lucara on actual achieved polished sales in excess of the initial estimated polished price, less a fee and the cost of manufacturing.

Throughout 2020 the Karowe Mine produced 779 specials that included 24 diamonds greater than 100 ct, including an unbroken 549 ct white diamond “Sethuyna” of exceptional purity and an unbroken 998 ct both from direct milling of EM/PK(S) South Lobe ore.

Work on the Karowe underground expansion project continued with an investment of $18.7 million under a re-scoped budget (due to COVID-19) that focused on de-risking the Project schedule (procurement of long lead equipment, detailed design and engineering) and minor early surface works.

In January 2021, Lucara announced that its application for the renewal of Mining License No 2008/6L in respect of the Karowe Mine has been approved by Botswana’s Minister of Mineral Resources, Green Technology and Energy Security. The renewal was effective January 4, 2021 for a period of 25 years, securing Lucara’s mining rights to 2046.

In April 2021, the HB sales agreement was extended for a 24-month period, effective from January 1, 2021 to December 31, 2022. Following the extension of the HB Agreement in Q2 of 2021, all +10.8 ct non-gem quality diamonds and all diamonds less than 10.8 ct in weight which did not meet the criteria for sale on Clara are being sold as rough through the quarterly tender process. In the agreement extension, changes to the payment terms were amended to better reflect the timing of mine production and the manufacturing process.

On July 12, 2021, Lucara Botswana, with Lucara as the sponsor and the guarantor, entered into a senior secured project financing debt package of $220 M with a syndicate of five mandated lead arrangers to fund the development of an underground expansion at the Karowe Mine refinance Lucara’s existing revolving credit facility and will be used to support on-going operations.

Two equity financings were closed in July 2021 that generated net proceeds of $31.3 M from the sale of 55,157,733 common shares at a price of C$0.75 per share, including the acquisition of 16.4 million common shares by Lucara's largest shareholder, Nemesia S.a.r.l. (Nemesia).

In September 2021, Lucara announced that the Karowe underground expansion project was formally approved by the Board of Directors after closing a $220 M senior secured project debt financing.

Throughout 2021-end Q1/23 open pit operations continued, and significant diamond recoveries of 1,174 ct in 2021 and a further 87 stones > 100 ct from milling of South Lobe ore.

Since the onset of commercial production to the end of Q2 2023, the Karowe Mine has produced 4.2 Mcts from 17 28 Mt of processed kimberlite and has sold via tender a total of 3.99 Mcts for a total of $2.2 B resulting in an achieved sold average price of $558/ct (Table 6-1).

Table 6-1: Karowe Mine Production and Sales Results

Year	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023*	Total
Kimberlite mined (tonnes)	1,600,971	3,944,343	3,327,754	2,358,657	2,722,375	1,575,052	3,113,362	3,303,375	2,987,775	3,668,677	2,497,119	1,224,036	32,323,496
Waste mined (tonnes)	4,074,196	5,493,445	10,270,720	11,407,010	11,058,041	15,865,121	15,001,820	6,542,781	2,662,327	2,619,744	1,493,113	1,668,346	88,156,664
Kimberlite processed (tonnes)	1,327,682	2,354,538	2,421,506	2,238,975	2,613,217	2,335,550	2,629,048	2,804,517	2,676,066	2,844,888	2,770,039	1,421,023	28,437,049
Carats recovered	294,167	440,751	430,292	365,690	353,974	249,767	366,086	433,060	381,707	369,390	335,768	180,137	4,200,789
Recovered grade (cpht)	22	19	18	16	14	11	14	15	14	13	12	13	14.8
Carats sold	152,724	438,717	412,136	377,136	358,806	260,526	350,799	411,736	372,941	380,493	327,027	156,091	3,999,132
Sales average $/ct	$274	$415	$617	$612	$824	$847	$502	$468	$334	$599	$623	$512	$558
	$41,846,376	$182,067,555	$254,287,912	$230,807,232	$295,656,144	$220,700,000	$176,200,000	$192,500,000	$124,600,000	$228,000,000	$203,800,000	$79,900,000	$2,230,365,219

Notes:
* To end June 2023

Source: Lucara (2023)

6.7.1 Significant Stone Recovery to June 30, 2023

From inception to the end of Q2 2023, the Karowe Mine has recovered 31 diamonds > 300 cts, 61 diamonds between 200 and 300 cts and an additional 229 diamonds between 100 and 200 cts. The mine has recovered three diamonds in excess of 1000 ct since 2015. Since 2012 the mine has produced over 244,000 ct and over 8000 stones in excess of +10.8 ct for an average stone size of 30.4 ct/stn for the +10.8 production. In the period 2015 to June 30,2023 inclusive the mine has averaged 6.4 wt% of total production being >10.8 ct in weight. During the period of Q3 and Q4 2023 Karowe produced an additional 7 diamonds > 100 ct in size, including a 1080 ct gem quality diamond, bring the total number of +1000 ct diamonds to 4 since 2015.

7 GEOLOGICAL SETTING AND MINERALIZATION

A detailed account of the geological setting and geology of the KDM was provided in Lynn et al. (2014). A summarized version from Nowicki et al. (2018) was restated in Doerksen et al. (2019) with additional information in Sections 7.3 and 7.4 documenting changes to the geological model, in particular for the deep portion (below ~500 masl) of the South Lobe, based on core drilling undertaken in 2018-2019.

7.1 Local and Regional Geology

KDM is exploiting the AK6 kimberlite which is part of the Orapa Kimberlite Field (OKF) in the Central District of Botswana. The OKF includes at least 83 kimberlite bodies of post-Karoo age. Three of these (AK1, BK9, and AK6) have been or are currently being mined and four (BK1, BK11, BK12 and BK15) are recognized as potentially economic deposits.

The country rock at KDM is sub-outcropping flood basalt of the Stormberg Lava Group, underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup, below which is the granitic basement. The Jurassic (180 Ma) basalts, which are very extensive and underlie much of central Botswana, lie unconformably on the sedimentary succession but are stratigraphically part of the Karoo Supergroup. The regional stratigraphy is shown in Table 7-1. Rocks close to surface are typically extensively calcretized and silcretized due to prolonged exposure on a late Tertiary erosion surface (the African Surface) which approximates to the present-day land surface. 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 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 but are seldom exposed within the Makgadikgadi Depression. The condensed Karoo succession has a total thickness of around 600 m and is best preserved in WNW-ESE oriented grabens. The AK1 kimberlite (Debswana's Orapa Mine) lies within such a graben (Coates et al., 1979).

Table 7-1: Regional Stratigraphy

Stratigraphic Unit			Lithologies
Supergroup	Group	Formation	
	Kalahari Group	Not differentiated in this area	Windblown sand, overlying duricrusts
~~*unconformity*~~~~~~~~~~~~~~~~~~~~~~			

Stratigraphic Unit			Lithologies
Supergroup	Group	Formation	
			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

Source: McGeorge et al. (2010)

7.2 Property Geology

Drilling has defined the country rock succession at the KDM property as shown in Table 7-2. The volcanic and sedimentary units are almost flat lying.

Table 7-2: Stratigraphic Thicknesses at the KDM Property

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

Source: modified after McGeorge et al. (2010)

7.3 Kimberlite Geology

The description of the AK6 kimberlite geology presented in Nowicki et al. (2018) was extracted and summarized from internal De Beers documentation (Hanekom et al., 2006; Stiefenhofer, 2007; Tait and Maccelari, 2008) and from a Mineral Services report (MSC18/005R) documenting core logging, review and petrography work conducted in 2017/2018. These summaries are restated here, with additional information presented for the South Lobe based on core logging and petrography undertaken by SRK (SRK, 2019) for the 2019 FS (Doerksen et al. 2019). SRK has not carried out core logging and petrography for the North and Centre Lobes.

AK6 is a roughly north-south trending elongate kimberlite body with a surface expression of ~3.3 ha and maximum area of ~8 ha at approximately 120 m below surface. It comprises three geologically distinct, coalescing pipes known as the North, Centre and South Lobes that taper with depth into discrete roots. The North and Centre Lobes taper quite sharply, whereas the South Lobe is more cylindrical at depth. The South Lobe is the largest of the three lobes and makes up the bulk of the resource. KDM is one of the world's most significant producers of large and high-value diamonds including Type IIa and coloured diamonds.

The kimberlite in each lobe is different, in terms of its textural characteristics, relative proportion of internal country rock dilution, degree of weathering and alteration, as well as the characteristics of mantle-derived components including the diamond populations (Section 14). The South Lobe is distinctly different from the North and Centre Lobes which are similar in terms of their geological characteristics. The South Lobe is broadly massive and more homogeneous than the North and Centre Lobes which exhibit greater textural complexity and more variable and higher proportions of internal country rock dilution.

The kimberlite in each lobe has been grouped into mappable units (Table 7-3) based on its geological characteristics and interpreted grade potential, including separation of material with very high-country rock xenolith dilution (historically referred to as breccias). This is based primarily on extensive drill core logging and core photo review, supported by petrographic studies of representative samples, as well as historical 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; SRK, 2019). The main geological features of each unit are summarized below. Unless otherwise stated, the kimberlite terminology and olivine and country rock xenolith size and abundance descriptors used are from Scott Smith et al. (2013, 2018). Note that historical unit names have been maintained for consistency with previous reporting. Minor new units identified in the South Lobe since 2017 are denoted by non-genetic, numbered codes (e.g., KIMB1).

Note that the upper calcretized and weathered horizons in each lobe (Section 7.3.1) have now been mined out. Zones of high-country rock dilution (breccias) are present in each lobe; they appear to be largely restricted to the upper weathered, now-depleted portion of the South Lobe, whereas in the Centre and North Lobes they extend to greater depths.

Table 7-3: Kimberlite Units Identified in the AK6 Kimberlite

Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
	CKIMB	CKIMB(N)	Calcretized 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
Centre	BBX	BBX(C)	Country rock breccia
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite
	CKIMB	CKIMB(C)	Calcretized 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)	Calcretized country rock breccia
	CKIMB	CKIMB(S)	Calcretized 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

Lobe	Unit	Domain	Description
	WK	WK(S)	Weathered kimberlite
	WM/PK(S)	WM/PK(S)	Western magmatic/pyroclastic kimberlite
	KIMB1*	n/a	Volumetrically minor hypabyssal kimberlite
	KIMB3	KIMB3	Minor hypabyssal kimberlite; increasing volume below 500 masl
	KIMB4a	EM/PK(S)	Localized variant of EM/PK(S)
	KIMB5*	n/a	Volumetrically minor hypabyssal kimberlite
	KIMB6*	n/a	Volumetrically minor hypabyssal kimberlite
	KIMB7*	n/a	Volumetrically minor kimberlite

Notes:
*Minor units are included in the major domain models; same applies to KIMB3 intersections not included in the KIMB3 domain.
Units occurring in more than one lobe (e.g., BBX, CKIMB, WK) are modelled as separate domains for each lobe (denoted by N, C or S suffix) in the geological model.

Source: SRK (2023)

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. The zones of very high-country rock dilution (note the historical term breccia has been maintained for continuity with previous reporting) comprise either brecciated country rock blocks with minor matrix kimberlite or zones of high xenolith content within the pipe. The calcretized, weathered and breccia units are described below. Note that the geological domain models representing these units have been separated by lobe (Table 7-3).

Calcretized Kimberlite (CKIMB)

The upper parts of all three lobes comprised severely calcretized and silcretized rock. This zone was 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 was 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 kimberlite in each lobe was highly weathered. The intensity of weathering decreased 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 were obscured in the upper portions of the weathered zone, this material was seen to transition into the underlying fresh kimberlite units in each lobe. Due to the impact of weathering on the metallurgical properties of kimberlite, separate weathered units were defined in each lobe for those domains where weathered equivalents of the domains were present at surface.

Basalt Breccia (BBX/KBBX)

Discontinuous zones of brecciated basalt (BBX), mixed with variable, but generally minor amounts of kimberlite (typically less than 10 %) occur in each of the lobes; they consist of large (metre-sized) to smaller basalt clasts set in a matrix of kimberlite and the majority occur close to the wall-rock contact. An additional unit (KBBX) was defined to encompass kimberlite breccias that are broadly similar to the BBX 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 as either talus-type slump deposits or as deposits of possible pyroclastic origin (given their higher kimberlite content relative to BBX). These are now mined out in the South Lobe but extend below the current mining level in Centre and North Lobes.

7.3.2 North Lobe Kimberlite Units

Fragmental Kimberlite - FK(N)

The North Lobe is predominantly infilled by light greenish-grey, fine- to coarse-grained olivine-rich, matrix-supported, poorly sorted, massive volcaniclastic (fragmental) to superficially coherent (historically magmatic) kimberlite (Hanekom et al., 2006). Basalt is the dominant country rock xenolith type with lesser basement and Karoo sedimentary rock xenoliths. Two broad textural groups were identified in the kimberlite of the North Lobe: 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 coherent (historically magmatic) material as well as volcaniclastic (fragmental) kimberlite (Hanekom et al., 2006). Macroscopically, colour and texture variations are common within the Centre Lobe, but contacts between texturally distinct zones are generally gradational. The kimberlite textures locally alternate between apparent coherent and volcaniclastic, similar to the North Lobe. Hanekom et al. (2006) noted that the most consistent recognizable 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 recognized in the Centre Lobe, as described below.

Carbonate-Rich Fragmental Kimberlite - CFK(C)

The fresh infill in the upper part of the Centre Lobe comprises a fine- to coarse-grained olivine-rich, matrix-supported, poorly sorted and massive, carbonate-rich volcaniclastic (fragmental) to apparent coherent (historically magmatic) kimberlite. Basalt is the dominant country rock xenolith type with lesser basement and Karoo sedimentary rock fragments. Microscopically, most samples show carbonate infilling of void space, highlighting the 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 the fragmental kimberlite unit (see FK(C) – Fragmental kimberlite below). The groundmass opaque-mineral content is also slightly higher, although overlap occurs.

Fragmental Kimberlite - FK(C)

The remaining fresh kimberlite within the Centre Lobe comprises matrix-supported, poorly sorted and massive volcaniclastic (fragmental) to apparent coherent (historically magmatic) kimberlite which is distinct from CFK(C) due to an apparent relative decrease in carbonate content. Basalt is the dominant country rock xenolith type with lesser basement and Karoo sedimentary rock xenoliths. 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 higher country rock xenolith content. This material is often greenish in colour and characterized by the presence of large blocks of basalt. Basalt breccia (BBX) units in the Centre Lobe occur within the fragmental kimberlite unit rather than in the carbonate-rich fragmental kimberlite unit.

7.3.4 South Lobe Kimberlite Units

The upper part of the South Lobe (~ 70 – 100 m thick zone) which was dominated by weathered kimberlite (WK(S)), a weathered basalt breccia (WBBX(S)), an underlying unaltered basalt breccia (BBX(S)) and a large block (floating reef) of solid basalt (INTSWBAS) mapped during mining activities in 2013 (Lynn et al., 2014) has now been mined out. In addition to these weathered and breccia units, two volumetrically dominant kimberlite units (M/PK(S) and EM/PK(S)) have been recognized, as well as a further six volumetrically minor units, one of which (KIMB3) becomes more prevalent with increasing depth in the pipe.

Descriptions of the M/PK(S), EM/PK(S), KIMB1 and KIMB3 units provided in Nowicki et al. (2018) are restated here with additional information based on 2018/2019 work by SRK which includes (i) variations observed in the main units at depth in the pipe, (ii) updated description of KIMB3 based on improved understanding of this unit from numerous new drill intersections, and (iii) description of three additional minor units identified since the last update. Description of the WM/PK(S) is unchanged from Oberholzer et al. (2017).

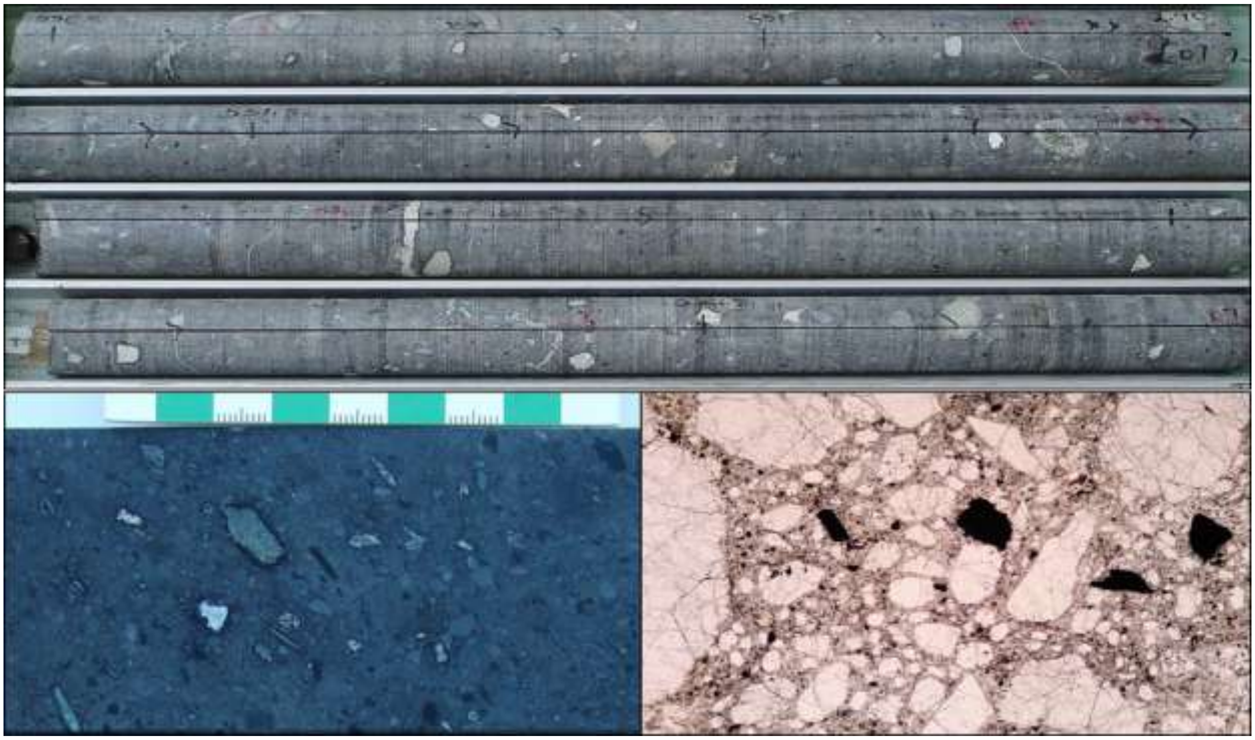
Magmatic/Pyroclastic Kimberlite - M/PK(S)

M/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly massive to locally crudely stratified macrocrystic apparent coherent kimberlite. In drill core, M/PK(S) is grey or grey-green in colour and exhibits a 'black spotted' appearance imparted by the presence of common completely kelyphitized (black/brown) garnet macrocrysts and black altered phlogopite macrocrysts. Crude stratification in the form of diffuse fluctuations in olivine and country rock xenolith size and abundance, and preferentially oriented elongate components (such as olivine, small basalt xenoliths, phlogopite macrocrysts) is variably developed. 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 (± serpentinized) monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. A distinct population of thermally metasomatized/ altered country rock xenoliths comprises mainly basalt (as larger grey-green clasts and small <1 cm white elongate shards), lesser (but visually distinctive) white basement granite/gneiss clasts with dark halos and minor Karoo sedimentary rocks. Total country rock dilution is typically low (<10 %), rarely ranging to a maximum of 25 %, and the majority of xenoliths are <10 cm in size. Ilmenite is notably abundant and characterized by variably developed grey reaction rims (comprising fibrous kelyphite-like material). In addition to garnet, ilmenite and rare chrome diopside, M/PK(S) contains orthopyroxene xenocrysts with variably developed reaction rims. The mantle mineral suite includes a distinct population of ultra coarse-grained (> 16 mm, with some up to 5 cm) garnet, ilmenite and orthopyroxene crystals which along with ultra coarse-grained olivine and phlogopite macrocrysts likely belong to the megacryst suite (Schulze, 1987). Peridotite and eclogite xenoliths are present throughout. M/PK(S) is characterized 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 crystals, combined with the presence of crude stratification and rare probable relict melt-bearing pyroclasts, suggest M/PK(S) was formed extrusively, and can be described as having a clastogenic or apparent coherent texture. Such kimberlites are believed to form by a range of processes which include lava fountain-type pyroclastic eruptions and effusive lava flows within an open diatreme or crater setting.

The name M/PK(S) applied to this unit reflects the historical uncertainty with respect to textural classification of the kimberlite - it exhibits textures consistent with magmatic (M), now referred to as coherent, kimberlite (Scott Smith et al., 2013), but also exhibits subtle textures suggesting a pyroclastic (P) origin. M/PK(S) is the volumetrically dominant South Lobe infill above ~550 masl. Typical M/PK(S) is shown in core, polished slab and photomicrograph in Figure 7-1.

Figure 7-1: Typical Appearance of M/PK(S)



Notes:
In HQ drill core (top, hole REP001 from 550 to 554 m), in polished slab (bottom left, hole REP002 at 639.81 m, cm scale) and in photomicrograph (bottom right, hole REP001 at 628.3 m, 20X magnification, PPL, FOV = 7 mm).

Source: Nowicki et al. (2018)

Eastern Magmatic/Pyroclastic Kimberlite - EM/PK(S)

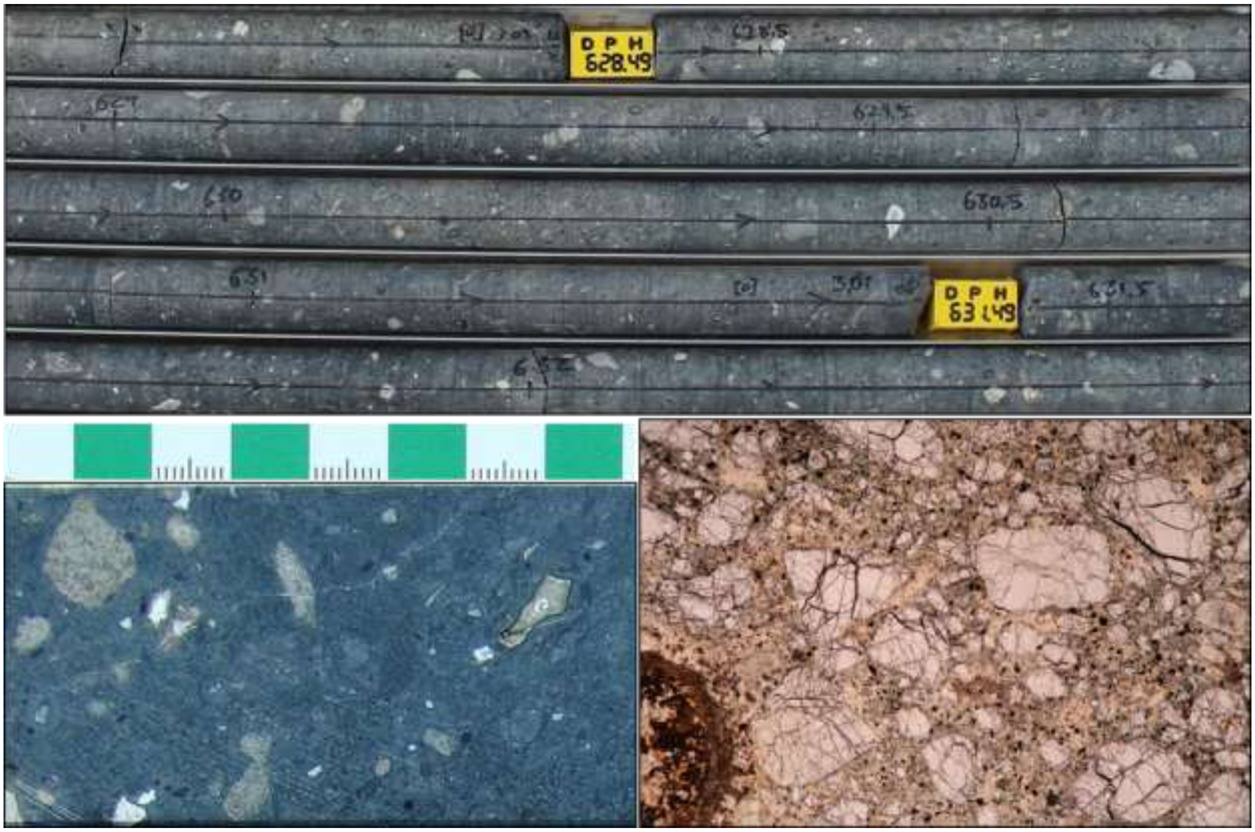
EM/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly massive to locally crudely stratified macrocrystic apparent coherent kimberlite. In drill core, EM/PK(S) is grey-green in colour with variably abundant white 'speckles'. 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 completely kelyphitized garnet is less common and phlogopite macrocrysts are fresh. Crude stratification in the form of diffuse fluctuations in olivine and country rock xenolith size and abundance is variably developed; 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 monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. Monticellite is typically serpentinized, but the proportion of fresh crystals gradually increases below ~500 masl, and below ~300 masl most samples comprise only fresh monticellite. Groundmass spinel is less abundant than in M/PK(S) and generally occurs as single crystals, with crystal aggregates being 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 xenoliths are more abundant than granite-gneiss xenoliths and more commonly exhibit zonal alteration and irregular clast margins. The small (<1 cm) white ‘speckles’ characteristic of this unit include round carbonate/clay-rich fragments that are possible amygdales derived from disaggregated basalt. The thermal metasomatism/ alteration assemblage of country rock xenoliths in EM/PK(S) includes common clinopyroxene. Total country rock dilution is typically low (<15 %), rarely ranging to a maximum of 25 %, and the majority of xenoliths are < 10 cm in size. As in M/PK(S), ilmenite is characterized 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. The mantle mineral suite similarly includes a distinct population of ultra coarse-grained (> 16 mm with some up to 5 cm) garnet, ilmenite and orthopyroxene crystals which along with ultra coarse-grained olivine and phlogopite macrocrysts likely belong to the megacryst suite (Schulze, 1987). Peridotite and eclogite xenoliths are present throughout. 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 crystals, combined with the presence of crude stratification and rare probable relict melt-bearing pyroclasts, suggest EM/PK(S) was formed extrusively, and can be described as having a clastogenic or apparent coherent texture. Such kimberlites are believed to form by a range of processes which include lava fountain-type pyroclastic eruptions and effusive lava flows within an open diatreme or crater setting.

As for M/PK(S) described above, the name EM/PK(S) applied to this unit reflects the historical uncertainty with respect to textural classification of the kimberlite - it exhibits textures consistent with magmatic (M), now referred to as coherent, kimberlite (Scott Smith et al., 2013), but also exhibits subtle textures suggesting a pyroclastic (P) origin. EM/PK(S), which historically was thought to occur only in the east (hence, E) of the pipe is the volumetrically dominant South Lobe infill below ~550 masl. Typical EM/PK(S) is shown in core, polished slab and photomicrograph in Figure 7-2.

A potential variant of EM/PK(S) referred to as KIMB4a is observed below ~500 masl as several dispersed drill intersections located close to or contiguous with M/PK(S) or KIMB3 or both. It differs from EM/PK(S) mainly in having a higher abundance of ilmenite, approximating that of M/PK(S). It is further distinguished by lower proportions of small basalt and Karoo sedimentary xenoliths, paucity/lack of clinopyroxene in xenolith alteration assemblages, more commonly altered phlogopite macrocrysts, generally higher groundmass spinel abundance and different degree/style of olivine alteration. The magnetic susceptibility of KIMB4a is at the high end of the range for EM/PK(S) (> 10 x 10^{-7} SI) and some values are as high as those for M/PK(S). Other features in the rock are consistent with EM/PK(S) and preclude a M/PK(S) classification.

Figure 7-2: Typical Appearance of EM/PK(S)



Notes:
In NQ drill core (top, hole GT001a from 628.0 to 632.5 m), in polished slab (bottom left, hole REP003 at 609.95 m, cm scale) and in photomicrograph (bottom right, hole REP003 at 588.58 m, 20X magnification, PPL, FOV = 7 mm).

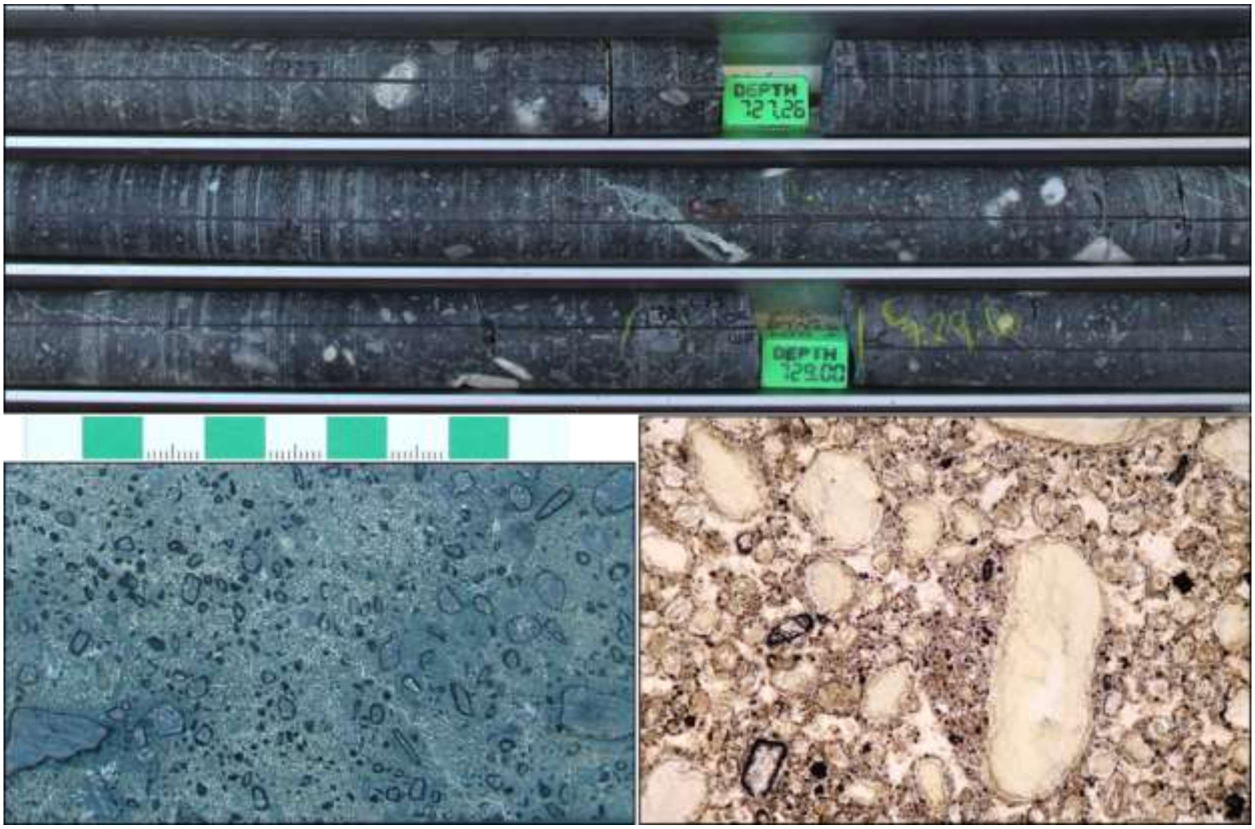
Source: Nowicki et al. (2018)

<u>Minor Unit KIMB3</u>

KIMB3 was identified during core logging and petrographic study undertaken in the South Lobe since 2017 (MSC18/005R; SRK, 2019). Although a volumetrically minor component (<5 %) of the total unweathered South Lobe infill, 2018-2019 drilling indicates it becomes more prevalent with depth in the pipe, particularly below 400 masl, where it occurs as numerous, closely spaced intersections alternating with intervals of (primarily) EM/PK(S). These "KIMB3-rich" areas have been modelled as a discrete geological domain (Section 7.3). Above ~550 masl, the more discontinuous and dispersed occurrences of KIMB3 (along pipe contacts, internal contacts and randomly within the main units) are not readily modelled as a separate domain and therefore have been incorporated into the surrounding M/PK(S) and EM/PK(S) domains in the geological model.

KIMB3 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor, massive macrocrystic hypabyssal kimberlite. In drill core, KIMB3 is dark grey-green in colour and characterized 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 M/PK(S), EM/PK(S) and KIMB1. The groundmass displays a variably developed segregationary texture and comprises acicular to prismatic decussate non-pleochroic phlogopite laths, serpentinized monticellite, perovskite, spinel (including common atoll textured crystals), serpentine/chlorite, carbonate and abundant hydrogarnet. Country rock dilution is typically very low (0-2 %) and the xenolith population comprises mainly basalt and granite-gneiss. Garnet is either partly fresh or completely kelyphitized and ilmenite variably lacks or has reaction rims like those observed in M/PK(S) and EM/PK(S). Garnet, ilmenite and mantle xenoliths are generally present in lower abundances than in the other units. Phlogopite macrocrysts are more common than in the other units and are typically completely altered. Autoliths of M/PK(S) and EM/PK(S) and others of unknown origin occur locally. Contacts between KIMB3 and M/PK(S) or EM/PK(S) are diffuse or sharp and finer-grained flow zones are commonly observed at contacts. Well-developed flow differentiation between finer- and coarser-grained components is observed in some intersections. Together these features suggest KIMB3 represents low-volume late-stage sheet intrusions emplaced into the main pipe filling units, possibly in some cases before they 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. Typical KIMB3 is shown in core, polished slab and photomicrograph in Figure 7-3.

Figure 7-3: Typical Appearance of KIMB3



Notes:
In HQ drill core (top, hole REP012 from 726.8 to 729.3 m), in polished slab (bottom left, hole REP012 at 729.53 m, cm scale) and in photomicrograph (bottom right, hole REP012 at 729.53 m, 20X magnification, PPL, FOV = 7 mm).

Source: SRK (2023)

Minor Unit KIMB1

KIMB1 was identified during core logging and petrographic study undertaken in the South Lobe since 2017 (MSC18/005R; SRK, 2019). It is a volumetrically minor component (<5 %) of the total South Lobe infill and generally occurs as discontinuous and dispersed occurrences along the pipe contacts, internal contacts and apparently randomly within the main units, in some cases spatially associated with KIMB3. It has not been modelled as a separate domain and is incorporated into the surrounding M/PK(S) and EM/PK(S) domains in the geological model.

KIMB1 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor massive to locally flow-aligned macrocrystic hypabyssal kimberlite. In drill core, KIMB1 is dark grey-black in colour with readily discernible mostly fresh 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) and the prismatic/acicular laths in KIMB3), lesser monticellite, perovskite, spinel, serpentine/chlorite and carbonate. Country rock dilution is typically low (<5 %) and includes basalt, granite-gneiss and Karoo sedimentary rock xenoliths in variable relative proportions. Both fresh and completely kelyphitized garnet are common and ilmenite generally lacks reaction rims like those observed in 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 between KIMB1 and 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 sheet intrusions emplaced into the main pipe filling units, possibly in some cases before they were completely consolidated. Magnetic susceptibility readings for KIMB1 are highly variable but most commonly < 20 x 10-7 SI.

Other Minor South Lobe Kimberlite Units

The three additional minor units identified since the last update, referred to as KIMB5, KIMB6 and KIMB7, make up a volumetrically minor component (<2 %) of the South Lobe infill.

KIMB5 occurs in the southeast of the pipe below ~370 masl and appears to have intruded EM/PK(S). It is a fine to coarse grained olivine-rich, very country rock xenolith-poor massive to locally flow-aligned macrocrystic monticellite phlogopite hypabyssal kimberlite. It superficially resembles M/PK(S) due to the presence of common small (<1 cm) white basalt xenoliths including elongate shards. It is distinguished from EM/PK(S) by higher abundances of groundmass phlogopite (as coarse poikilitic plates) and groundmass spinel, and lower abundances of garnet, ilmenite and orthopyroxene.

KIMB6 occurs as dispersed thin intervals below ~ 280 masl and appears to have intruded EM/PK(S). It is a fine to coarse grained olivine-rich, very country rock xenolith-poor massive macrocrystic phlogopite monticellite hypabyssal kimberlite. It superficially resembles M/PK(S) due to the presence of common small (<1 cm) white basalt xenoliths including elongate shards. It is distinguished from EM/PK(S) by a different olivine population and lower ilmenite abundance.

KIMB7 occurs along the pipe contact with the thickest intersections below ~120 masl. It is broadly similar to EM/PK(S) and is distinguished mainly by significantly lower abundances of garnet, ilmenite and orthopyroxene and by different relative proportions of country rock xenolith types, having more common basement granite and carbonaceous mudstone.

Western Magmatic/Pyroclastic Kimberlite - WM/PK(S)

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, fine to coarse grained, matrix-supported, poorly sorted, massive apparent coherent kimberlite (historically unclear if magmatic or pyroclastic) 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 serpentinized and locally completely weathered out from drill core. The WM/PK(S) is internally complex, both texturally and in terms of variability in country rock xenolith abundance, which ranges from <10 to 40%. Basalt is the dominant country rock lithology and ranges widely in size from < 1 to > 100 cm. Less common basement and rare black shale xenoliths are also present in places. The geometry of this unit is somewhat speculative due to sparse drill coverage. A possible additional WM/PK(S) intersection was obtained in 2018-2019 drilling which petrographically is similar to KIMB3, suggesting WM/PK(S) may be the near-surface product of KIMB3 observed at depth, or another similar phase of kimberlite.

7.4 AK6 Geological Model

The geological model of AK6 consists of two components: (1) a pipe shell model defining the geometry and extent of the deposit, and (2) an internal geological domain model comprising multiple wireframe solids that represent the spatial distribution of the various kimberlite and other (e.g., basalt breccia) units. The geological model was generated using Seequent's Leapfrog Geo software.

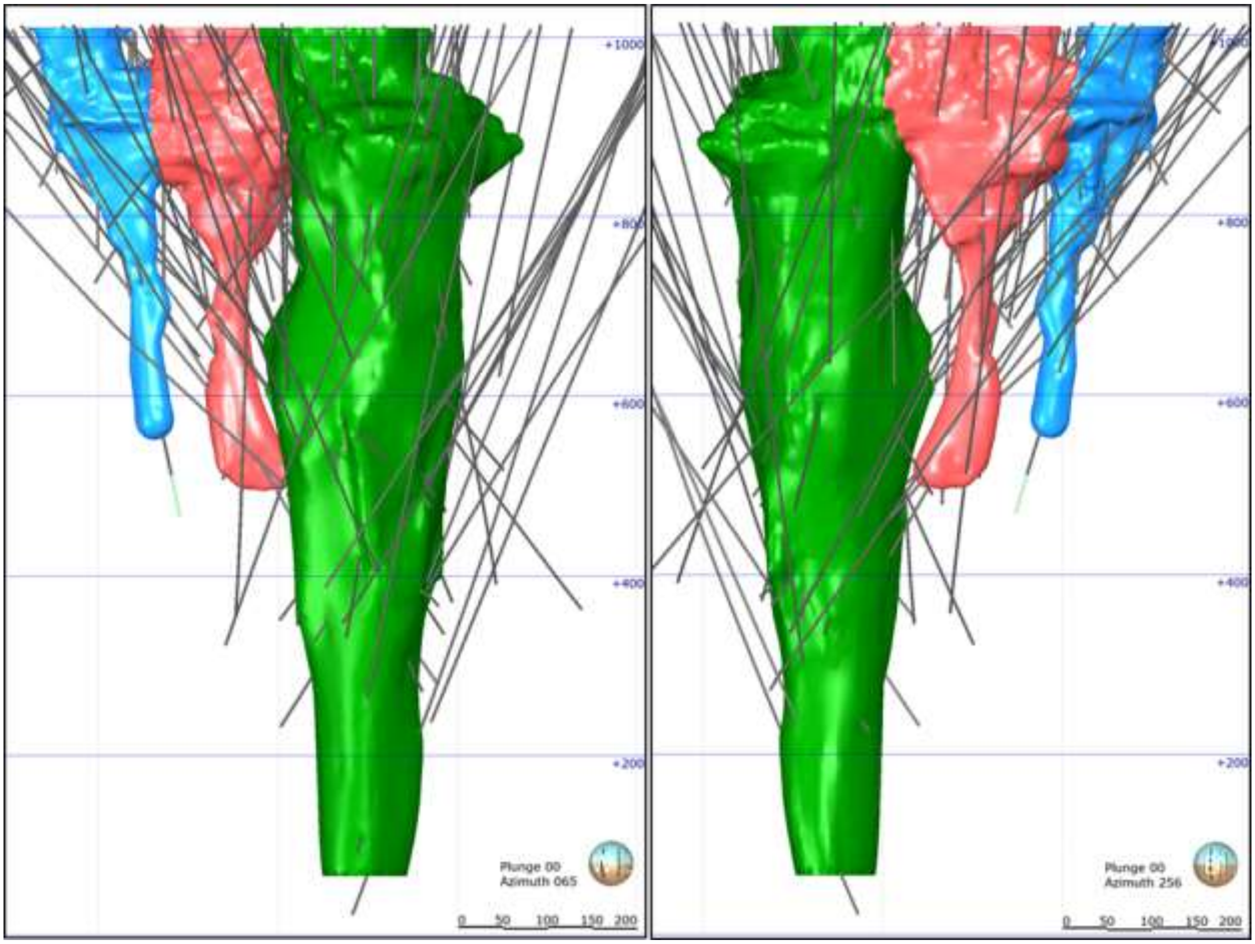
The pipe shell model was updated in 2019 (SRK, 2019; Doerksen et al., 2019) for mining exposure of the contact (all lobes) and at depth in the South, Centre and North Lobes using new pierce points from the 2018-2019 core drilling program. The base of the South Lobe model was extended by an additional 190 m. The internal domain model for the South Lobe was also revised based on logging and petrography of the 2018-2019 drill cores (SRK, 2019; Doerksen et al., 2019. The two main updates made in 2019 were: (1) a change in shape and decrease in size of the M/PK(S) domain below 500 masl and (2) generation of a new domain solid representing the distribution of the KIMB3 unit below 550 masl. No additional updates have been made since 2019. The internal domain model for the Centre and North Lobes remains unchanged from that documented in Oberholzer et al. (2017).

7.4.1 Shell Model

In 2019 the pipe shell model was updated for mining gains in all three lobes. In the South Lobe, the mapping data defined a pronounced 'bulge' in the pipe margin mainly in the southwest and southeast between 80 and 130 m below surface (920 to 870 masl), roughly corresponding with the contact between Stormberg basalt and Ntane sandstone wall-rocks. The downward extent of the gain is constrained by drilling. In the Centre and North Lobes, the volume increases occur from 70 to 100 m below surface (930 to 900 masl) mainly in the east and are similarly constrained below by drilling. These zones are now mined out.

The pipe shell model (all lobes) is defined by a total of 167 pierce points in 96 core drillholes and an additional 15 pierce points in 13 LDD holes. The South Lobe alone is defined by 87 pierce points in 56 core drillholes and 5 pierce points in 7 LDD holes. The 2018-2019 core drilling provided an additional 24 pierce points in 13 core drillholes in the South Lobe, ten of which occur below 400 masl. The substantial internal and external (country rock only) drill coverage provides additional guidance on the minimum and maximum shell constraints respectively. The South Lobe model extends from surface (~1000 masl) to a minimum elevation of 66 masl (Figure 7-4). The 2018-2019 core drilling supported extension of the base of the model by an additional 190 m (from 256 to 66 masl). The degree of control on the pipe shell is relatively high down to 250 masl, below which the model is based on only four pierce points and downward continuation of the established pipe contact dip (refer to Section 7.4.4). The North and Centre Lobe models extend to minimum elevations of 550 masl and 500 masl respectively.

Figure 7-4: AK6 Pipe Shell Model



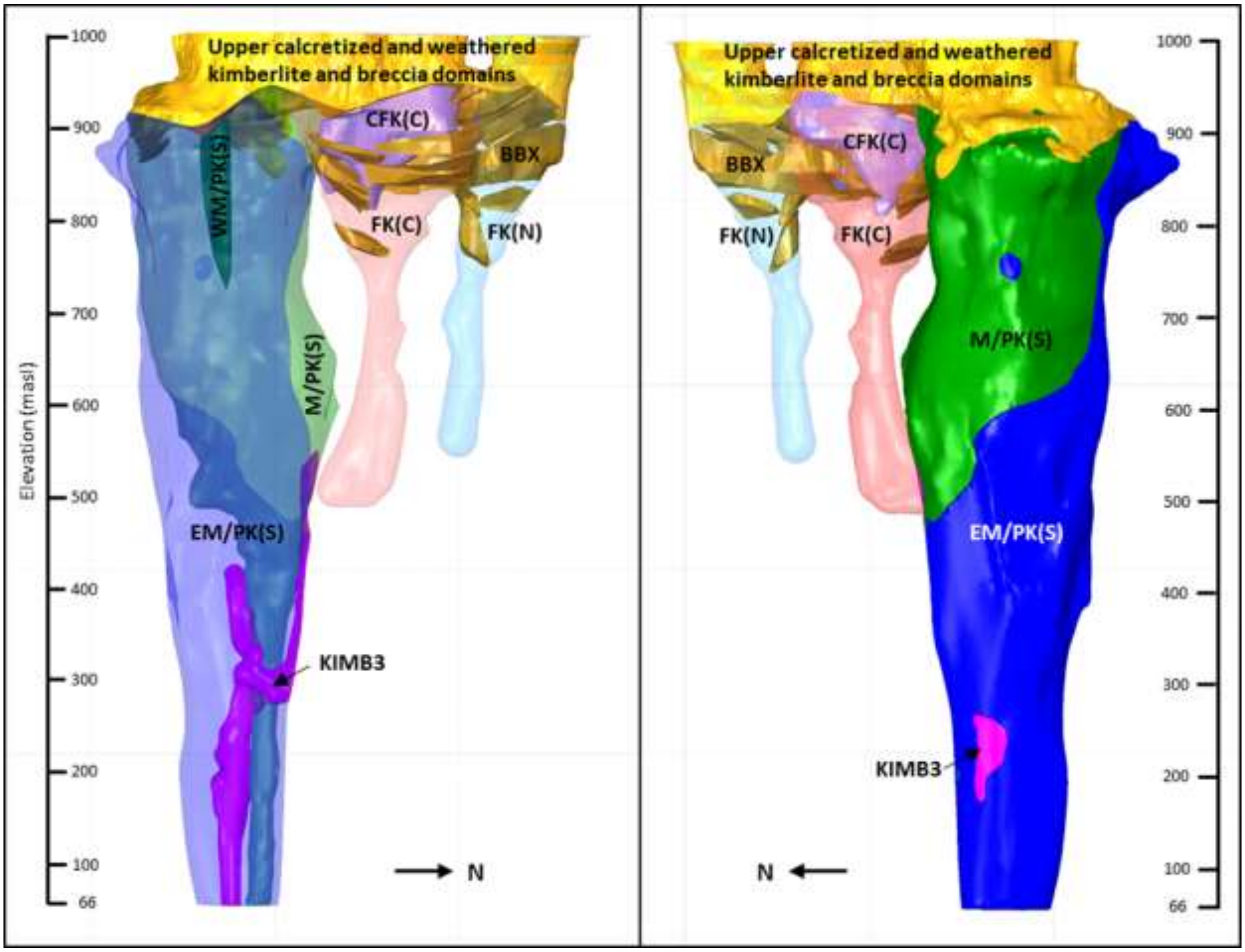
Notes:
Colour coded by lobe (blue = North, red = Centre, green = South) and showing all drillholes (black traces) used to define the model.

Source: SRK (2023)

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 and other (e.g., basalt breccia) units within each lobe (Table 7-3). The internal geological domains are shown in Figure 7-5 and the number and length of core drillholes defining each domain are given in Table 7-4.

Figure 7-5: Internal Geological Domains of the AK6 Kimberlite



Notes:
The upper ~70 to 100 m of calcretized and weathered kimberlite and country rock breccia units which are now mined out (July 1. 2019 pit surface ranges 115 to 155 mbs) are shown in a single colour to simplify the figure. Some domains are rendered transparent to display the internal domains.

Source: SRK (2023)

Table 7-4: Core Drill Coverage of Internal Geological Model Domains

Lobe	Domain	Number of Core Holes	Drillhole Intersection Length (m)
North	BBX(N), CKIMB(N), WBBX(N), WKBBX(N), WK(N)	13	914.6
	FK(N)	14	1,008.4
Centre	BBX(C), CKIMB(C), KBBX (C), WK(C)	20	1,264.9

Lobe	Domain	Number of Core Holes	Drillhole Intersection Length (m)
	CFK(C)	18	1,047.7
	FK(C)	25	1,272.0
South	BBX(S), CKBBX(S), CKIMB(S), WBBX(S), WKBBX(S), WK(S), IntSWBas	31	2,023.4
	M/PK(S)	52	8,201.3
	EM/PK(S)	44	5,038.1
	KIMB3	7	381.9
	WM/PK(S)	5	341.4

Source: SRK (2023)

In the South Lobe, the distribution of the two major kimberlite units, M/PK(S) and EM/PK(S), is represented by two separate domains. Most minor kimberlite units (and subunits/variants of the major units) have not been resolved as discrete domains (generally due to their discontinuous distribution) and these are included in the main domains, the exception being KIMB3 for which a separate solid has been generated in the updated model as explained below.

The M/PK(S) and EM/PK(S) model solids were updated in 2019, the most significant changes being below 500 masl. Above this elevation, the 2018-2019 drilling indicates a slight increase in the EM/PK(S) domain in the northeast of the pipe and the presence of minor EM/PK(S) along the southwest margin (previously not intersected in this area). Below 500 masl, the 2018-2019 drilling indicates a decrease in the modelled extent of M/PK(S) in the central part of the pipe where its southern boundary pinches sharply towards the north, with a corresponding expansion of the EM/P(KS) domain. Nowicki et al. (2018) noted that the M/PK(S) domain was poorly constrained by drilling below 450 masl and this remains the case in the updated model. The revised M/PK(S) domain model is not directly drill-supported below ~440 masl, other than by a short (~6 m) intersection at ~ 305 masl; however, the relatively common drill intersections of EM/PK(S) and KIMB3 above 300 masl provide maximum constraints on its extent (Figure 7-6). Below ~440 masl, the M/PK(S) domain has been modelled based on (i) an emplacement model for the South Lobe kimberlite which interprets the existence and likely preservation (within the earlier-emplaced EM/PK(S) infill) of a conduit for the large-volume M/PK(S) infill that dominates the upper part of the pipe, (ii) occurrence of the short M/PK(S) drill intersection at ~305 masl, and (iii) application of a conservative approach to modelling of the internal geology which takes into consideration the lower diamond grade and value of the M/PK(S) compared to the EM/PK(S) (Section 14).

The 2019 FS model update included the generation of a new model solid representing the areas where drilling to date suggests the KIMB3 unit is most common. As described in Section 7.3 above, KIMB3 is a hypabyssal kimberlite that post-dates and intruded into the M/PK(S) and EM/PK(S) kimberlites. KIMB3 occurs above 550 masl in both domains but becomes more prevalent below this depth, particularly below 400 masl in the central-west portion of the pipe where numerous KIMB3 intrusions occur within mainly EM/PK(S). These “KIMB3-rich” areas form the basis of the KIMB3 domain model, and the largest drill-defined portions have been connected based on an emplacement model that interprets KIMB3 as multiple generally vertically oriented late-stage sheet intrusions.

The volumes of the M/PK(S), EM/PK(S) and KIMB3 domains in various depth intervals are shown in Table 7-5. The morphologies of the domains and the internal drill coverage on which they are based are illustrated in Figure 7-6. No changes were made in 2019 to the internal domain boundaries reported in Oberholzer et al. (2017) for the North and Center Lobes, or for the South Lobe within the upper weathered/diluted zone (now mined out).

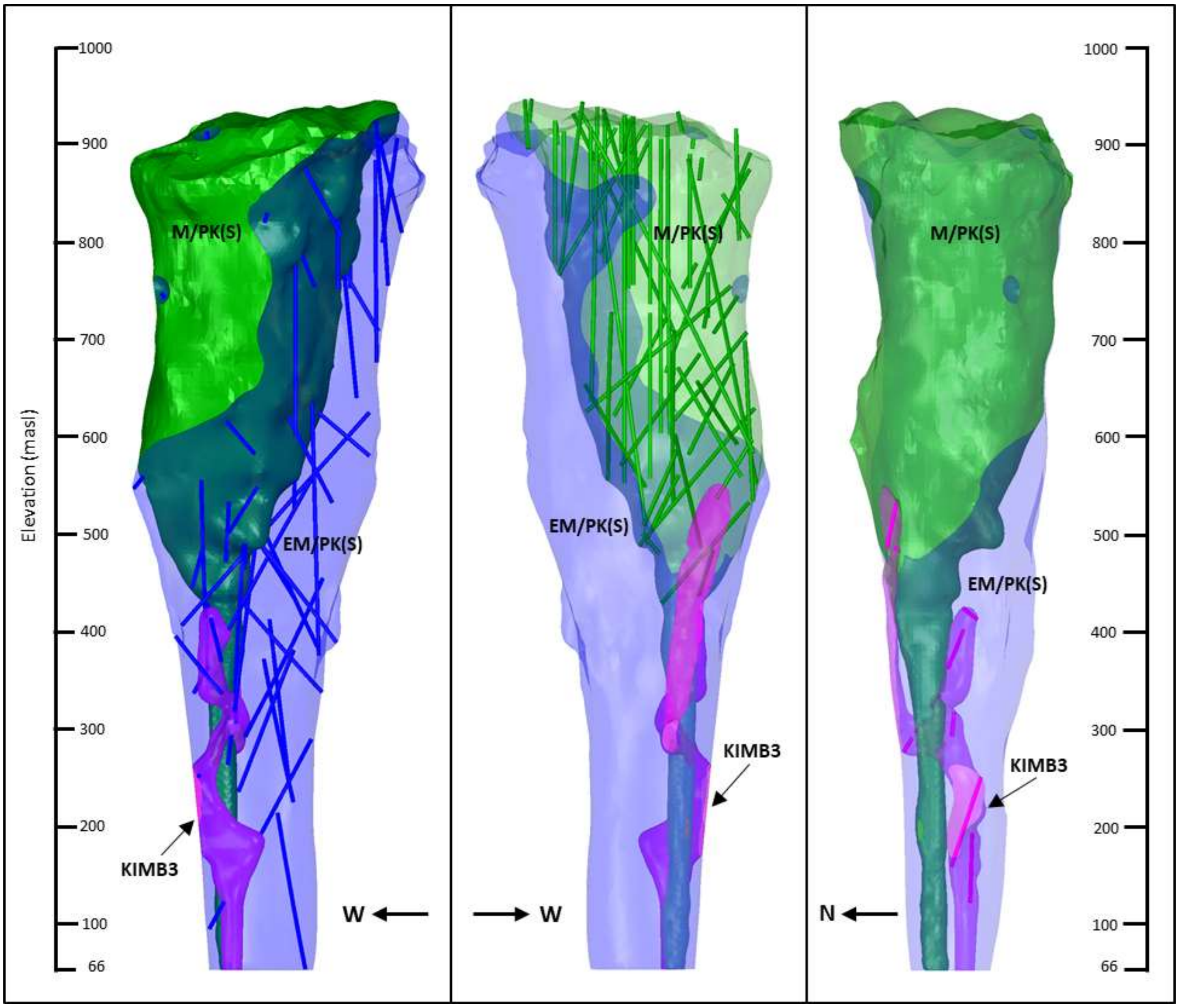
Table 7-5: Volume Estimates of South Lobe Internal Domains in Various Elevation Ranges (below June 30, 2023 pit surface)

Elevation range (masl)	All Domains	M/PK(S)		EM/PK(S)		KIMB3	
	Mm^3	Mm^3	%	Mm^3	%	Mm^3	%
Pit Surface (June 30, 2023) to 400	11.95	6.91	58	4.99	42	0.05	0.3
400 to 250	2.02	0.11	5	1.78	88	0.13	7
250 to 66	1.65	0.10	6	1.41	85	0.13	8
Total	15.62	7.12	46	8.18	52	0.32	2

Note:
Due to rounding some columns or rows may not compute exactly as shown.

Source: SRK (2023)

Figure 7-6: South Lobe Internal Domain Model



Note:
Looking north (left), south (middle) and east (right) showing the morphology of the M/PK(S), EM/PK(S) and KIMB3 domains (rendered transparent) and the internal core drill coverage used to define them.

Source: SRK (2023)

7.4.3 Geological Continuity

Demonstration of geological continuity within the main kimberlite units is required for the Mineral Resource Estimate to permit (1) assignment of average diamond values derived from production data to kimberlite at depth and (2) assignment of average grade estimates below 604 masl (Section 14). A thorough assessment of the degree of geological continuity was carried out by

MSC in support of the resource update reported in Nowicki et al. (2018). This involved review of surface exposures, drill cores and dilution measurements, and an extensive petrographic study. As described in Nowicki et al. (2018) and summarized below, this work confirmed that, with the exception of local variations in the amount of country rock dilution for the FK(C) and FK(N) units, the main kimberlite units in AK6 are internally broadly homogeneous. Ms. Webb of SRK carried out much of this work while employed at MSC and subsequently further assessed the degree of continuity within the kimberlite units based on work conducted since then.

Surface and Drill Core Observations

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). Kimberlite exposures in the OP were examined in July 2013, October 2013, June 2017, June 2018 and May 2019. A detailed review of ten complete drill cores was undertaken on site in 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, and a detailed review of 13 of the 2018-2019 drill cores was undertaken on site in May 2019. 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.

Internal Dilution

Line-scan measurements of country rock xenolith content provide a reliable broad-scale assessment of the dilution characteristics of the major kimberlite units. Data collected during historical, and 2017 core drilling suggest minor local variation and no significant large-scale dilution trends with depth in the main kimberlite units in the South Lobe. This is corroborated by data collected for 2018-2019 drillholes intersecting the deeper portion of the South Lobe (below 400 masl). The amount of dilution present in FK(C) and in FK(N) is on average approximately double that of the M/PK(S) and EM/PK(S) and is more variably distributed. Potential grade variation associated with variation in dilution in FK(N) and FK(C) is accounted for in the local grade interpolation method used for these units (Section 14).

Drill Core Petrography

A large suite of spatially representative petrography samples (n = 227) was collected from drill core in 2017 (92 from historical holes and 135 from 2017 deep drillholes). A further 128 petrography samples were collected from the deep 2018-2019 drillholes. The main objective of the petrographic analysis was to assess the degree of continuity with depth in M/PK(S) and EM/PK(S), the two major units of the South Lobe. 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 work indicated common small-scale variability in these parameters in the M/PK(S) or EM/PK(S), and the presence of a localized potential variant of EM/PK(S); it did not, however, reveal evidence for large-scale variations or trends in any of these parameters within the M/PK(S) or EM/PK(S) (MSC18/005R; SRK, 2019). Line-scan measurements of olivine size and abundance were not undertaken due to the observed broad-scale homogeneity in these parameters.

7.4.4 Confidence of Geological Model (Volume Estimate)

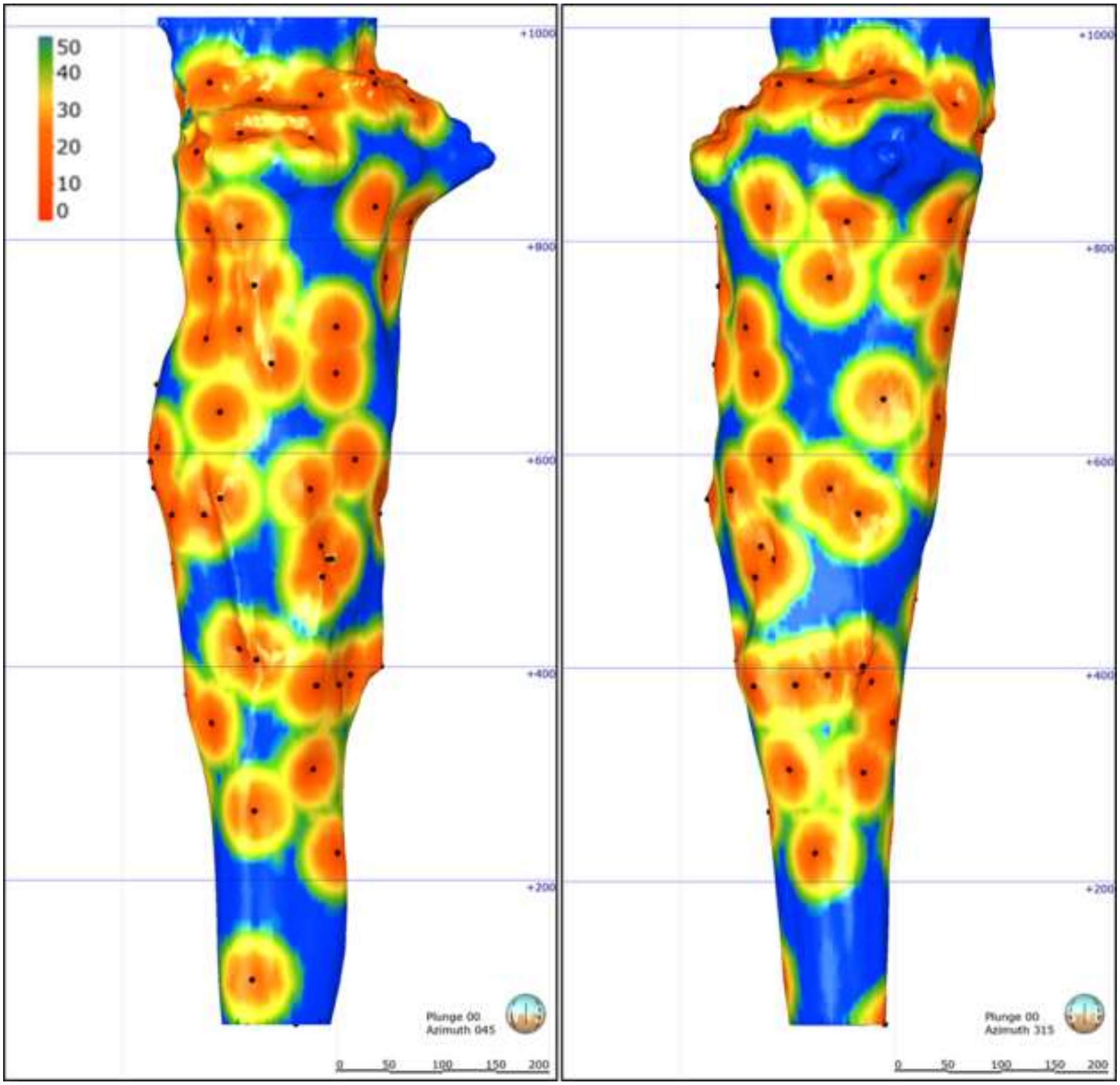
The AK6 pipe shell model is constrained by 182 pierce points from 109 core and LDD drillholes, the majority of which intersect above 600 masl. The model is well constrained in this upper zone by these pierce points and extensive internal coverage providing minimum constraints on the size of the body.

The South Lobe shell model is well constrained by 48 pierce points above 600 masl and by 23 pierce points between 600 and 400 masl. The 2018-2019 drilling provided an additional 14 pierce points in the South Lobe above 400 masl. The model is less well constrained by 12 pierce points between 400 and 250 masl, including six added by the 2018-2019 drilling. However, while there is scope to modify the exact position of the contact in the gaps between pierce points in this elevation range (Figure 7-7), it is unlikely that the overall pipe volume could deviate by more than ±10 % from the modelled estimate, based on (i) the high degree of confidence with which the shell is constrained above 400 masl and the good continuity with depth in the well-established side-wall dip as confirmed by deeper pierce points, and (ii) the reasonable internal coverage in this elevation range providing minimum constraints on the pipe volume. It is noted that the 20 pierce points added by the 2018-2019 drilling above 250 masl resulted in <1 % difference in volume between the 2018 (Nowicki et al., 2018) and 2019 (Doerksen et al., 2019) models in the zone below the July 1, 2019 pit surface and above 250 masl (i.e., excluding the mining gains realized between December 31, 2017 and July 1, 2019). Only four pierce points occur below 250 masl and there is consequently a higher degree of uncertainty in the pipe volume at this level.

The AK6 internal geological domain model is constrained by 21,494 m of internal core drilling, of which 15,986 m occurs in the South Lobe. The degree of control on the boundaries between the South Lobe internal domains is relatively high between surface and ~450 masl. There is only a single intersection of M/PK(S) below 440 masl and its volume is thus largely constrained by reasonable internal drill coverage, including intercepts of EM/PK(S) and the newly defined KIMB3 domain, which confirm where MP/K(S) is not present. The currently modelled distribution of KIMB3 likely represents a minimum volume for this unit.

Nevertheless, the uncertainty in Mineral Resource Estimates below 400 masl noted by Nowicki et al. (2018), which were 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, were significantly reduced by the 2018-2019 drilling. The additional drill coverage and microdiamond sampling provide a basis for upgraded confidence between 400 and 250 masl, excluding the KIMB3 domain (as noted in Section 14).

Figure 7-7: Drillhole Pierce Points in the South Lobe



Note:
Drillhole pierce points (black dots) in the South Lobe (left, looking northeast; right, looking northwest) with distance contours. Blue areas are > 50 m from pierce points.

Source: SRK (2023)

7.4.5 Summary

A considerable amount of drilling, geological logging and petrographic work has been undertaken at KDM in support of kimberlite geology development, resulting in a relatively high confidence geological model, which in the case of the South Lobe extends from surface to 250 masl.

8 DEPOSIT TYPES

This section is taken from Nowicki et al. (2018). 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 versus hard granite);
- Local structural setting;
- Local and regional stress field; and
- Presence of water.

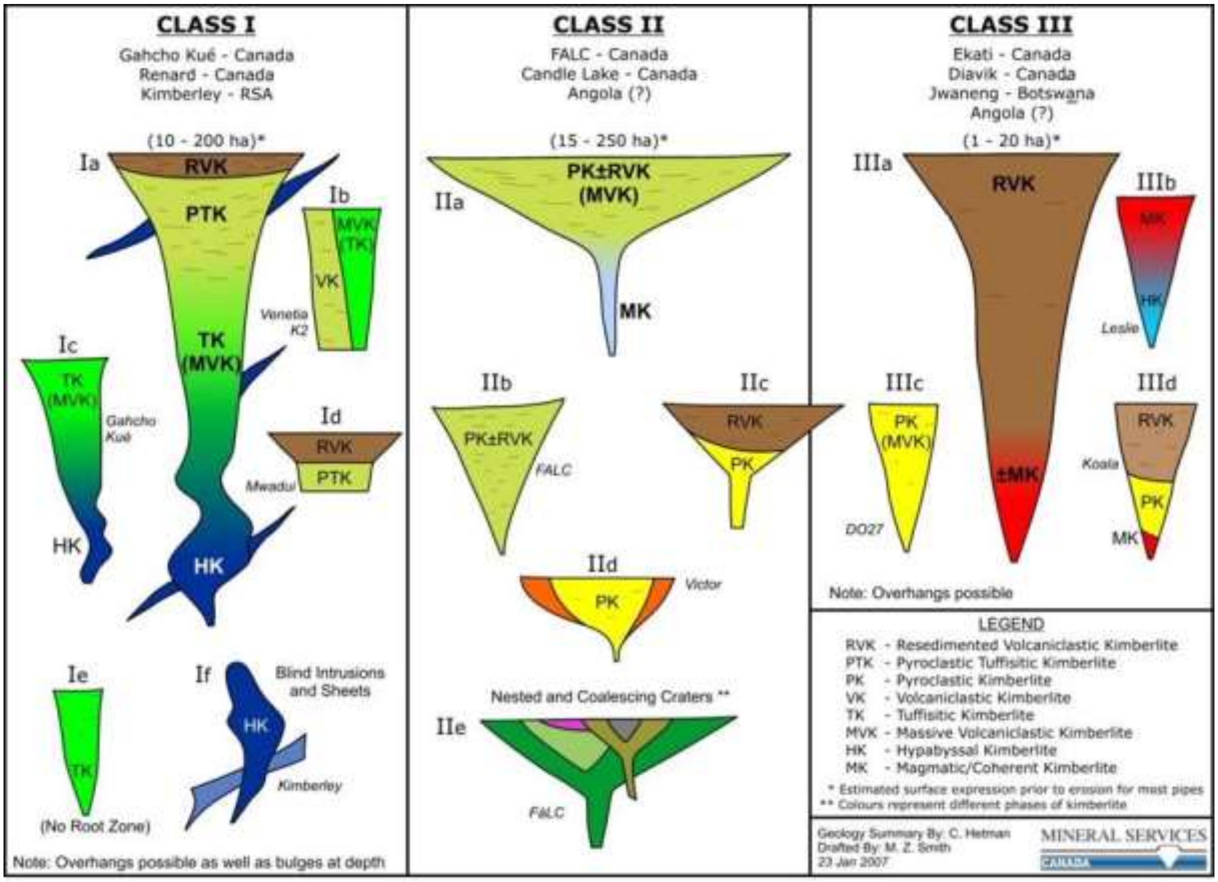
Kimberlites occur at surface as either sheet-like intrusions (dykes or sills) or irregular shaped intrusions and volcanic pipes. The sheets and irregular intrusions are typically emplaced along pre-existing planes of weakness in the country rock. Their emplacement does not involve explosive volcanic 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*



Notes:
*****The three classes (I, II and III) represent broad groupings with shared attributes of geometry, size and infill.

Source: Nowicki et al. (2018)

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; and

- 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 PL 13/2000. Details on previous work programs are briefly summarized here (extracted and summarized from Nowicki et al. 2018, Oberholzer et al., 2017) and are detailed in Lynn et al., 2014, McGeorge et al., 2010 and various references therein. Recent exploration completed in 2017-2019 included core drilling and sampling of core material and this 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, Nowicki et al., 2018).

The AK6 kimberlite was continuously held by De Beers under a succession of prospecting licenses from the time of its discovery in 1969, until the Project was acquired by Lucara in 2009. 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 the AK6 kimberlite is shown in Table 9-1. It 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 five 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 ten-month lapse between the completion of drilling and the release of the sampling results, De Beers committed PL 13/2000 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 an Inferred Mineral Resource and recover 500 cts from 13 large diameter drillholes at 70 m spacing. The external contacts and internal geology of the kimberlite were explored through an extensive program 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 cts, to reduce revenue uncertainty. Large diameter drillholes were placed at 50 m centres and trenches were prepared for recovery of the required parcel of diamonds. Further delineation drilling was also completed. 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 OP mining based on an initial economic assessment;

- In 2016 and 2017, two core drilling programs were conducted on the AK6 kimberlite. The combined 12,272 m drilled provided additional pierce points and geological information for the deeper portion of the South Lobe; and

- In 2018 and 2019, a combined geotechnical and delineation drill program was conducted with 35 drillholes for total metres drilled of approximately 22,000 m. Some drilling was specific to the country rock and several holes were designed to test the South Lobe geotechnical purposes with two holes specifically designed to test the South Lobe at depths below 400 masl.

Table 9-1: Summary of Major Exploration Phases at AK6

Stage	Work Done	Duration
Early Evaluation	5 x 12¼" large diameter drillholes totalling 679 m, 97 t bulk sample	2003 - 2005
	DMS and diamond recovery	
	Geophysical surveys	
Phase 1 Advanced Exploration	44 x 6½" percussion holes for delineation totalling 4,575 m	2005 - 2006
	12 x cored boreholes (NQ) as LDD pilots, totalling 2,980 m	
	17 x inclined boreholes (NQ) for delineation totalling 6,904 m	
	13 x 23" LDD totalling 3,699 m	
	DMS processing and diamond recovery from 1,775 t	
Phase 2 Advanced Exploration	11 x cored boreholes (NQ) as LDD pilots totalling 4,181 m	2006 - 2008
	29 x inclined boreholes (NQ) for delineation totalling 8,679 m	
	12 x 23" LDD totalling 4,265 m	
	Trench bulk sampling at surface	
	DMS processing and diamond recovery from 2,235 t	
Delineation And Geotechnical Drilling	15 x cored borehole (HQ and NQ) totalling 12,272 m	2016 - 2017
	916 microdiamond samples (7,315 kg)	
Delineation And Geotechnical Drilling	37 x cored boreholes (HQ and NQ) totalling 23,958 m	2018 - 2019
	153 microdiamond samples (1232.8 kg)	

Source: Lucara (2019)

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 was 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.

Source: Lucara (2019)

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 March 25, 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:

- Core drilling to delineate the extent of the kimberlite and to map its internal geology / density;
- Large diameter drilling (LDD) to obtain large kimberlite samples to support estimates of diamond grade and value; and
- Pilot core drilling adjacent to LDD holes confirm the geology and kimberlite units sampled.

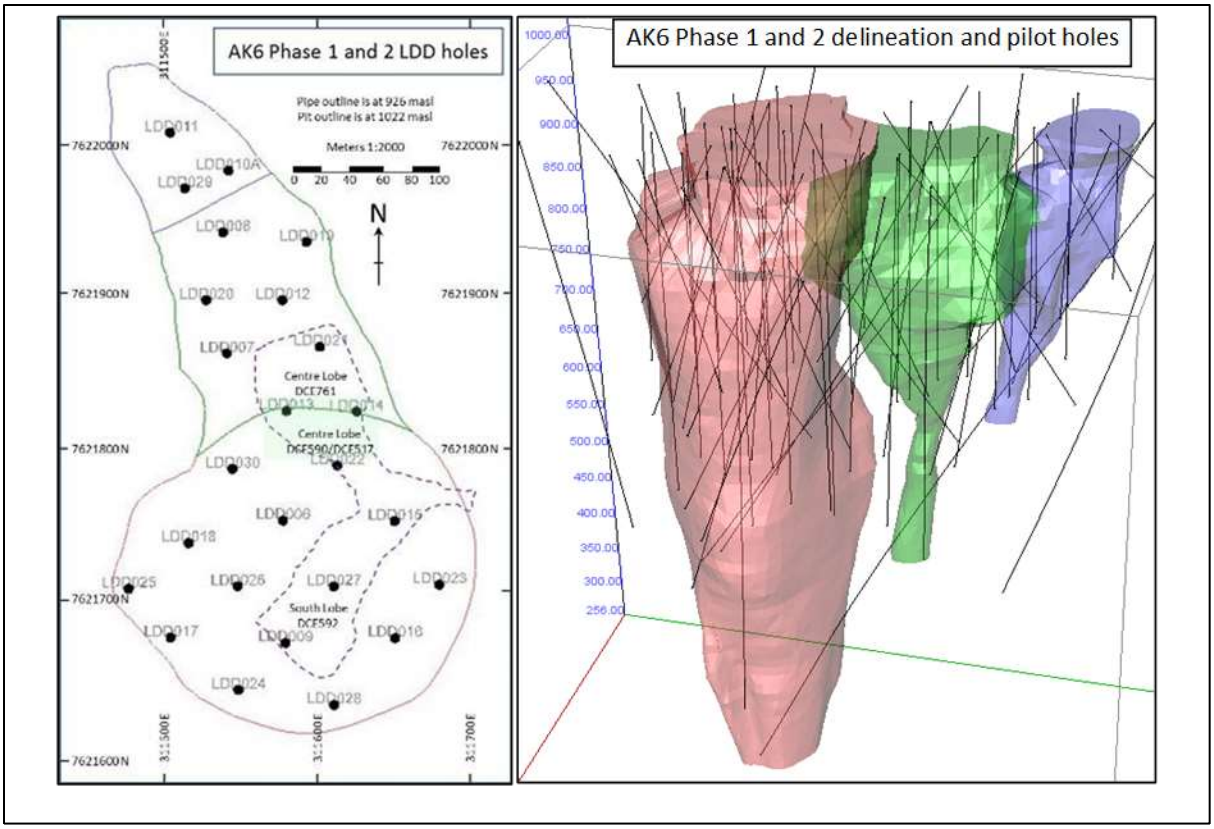
Drilling is summarized in Table 10-1, grouped into the exploration phases described in Section 9 above. Drillhole locations are illustrated in Figure 10-1.

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

Source: Lucara (2019)

Figure 10-1: AK6 Phase 1 and 2 Drillholes



Source: Lucara (2019)

Early evaluation holes are not shown as they were not used to support Mineral Resource Estimates. Large diameter Reverse Circulation (RC) holes (left, plan view) are all vertical, the outline of a surface trench bulk sample is shown as a dotted black line. Core drillholes (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

10.2.1 2017 Drilling 400-600 masl Definition

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 was completed from 15 drillholes, as summarized in Table 10-2. Drill coverage is shown in Figure 10-2. For certain holes survey of azimuth and dip could not be completed (five holes) to the base of the hole due to hole collapse and compression. Survey of azimuth and dip also produced highly irregular results in two holes. These drillholes with unreliable survey data were not used to support geological modelling.

Table 10-2: Recent (2017) Delineation (REP) and Geotechnical (GT) Drilling

Drillhole	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**			

Source: Lucara (2019)

10.2.2 2018 Drilling 250-400 masl Definition

During 2018 and 2019, a total of 37 core holes were drilling for geotechnical and delineation purposes (Table 10-3). The drilling provided geological information below 400 masl within the South Lobe to support further evaluation and geotechnical data (KGR series). Drilling was also conducted to provided geotechnical information on host rock stratigraphy (CR- GT series) and geotechnical data on potential UG infrastructure (INFRA series). Drill coverage for holes in 2017, 2018 and 2019 is shown in Figure 10-2.

Table 10-3: 2018 and 2019 Delineation (KGR) and Geotechnical Drilling (CR-GT, INFRA) Drilling

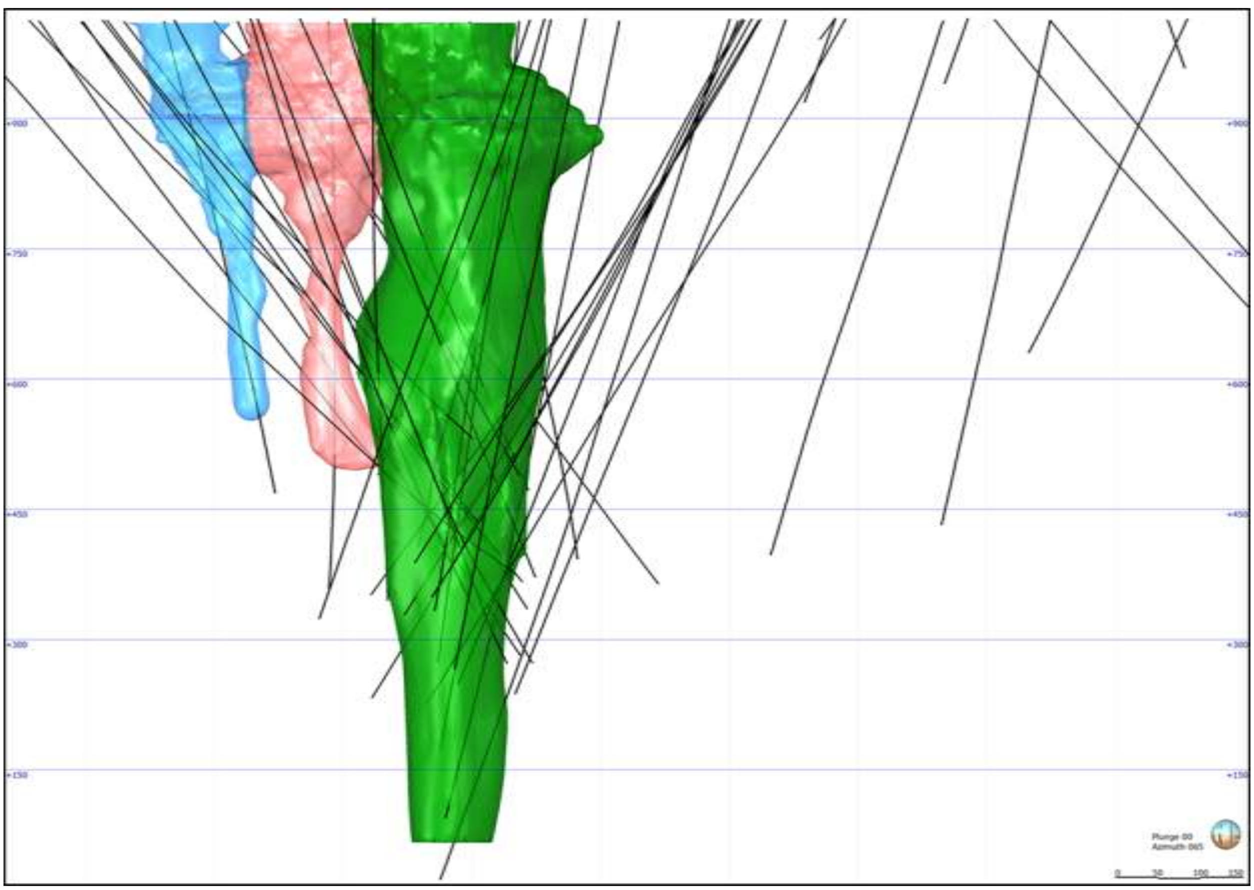
Drillhole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
CR_GT_DD001	341266	7621936	1013	876	113	-51
CR_GT_DD002	341379	7622174	1012	462	140	-44
CR_GT_DD003	341740	7622103	1012	900	189	-46
CR_GT_DD004	341944	7621869	1012	860	233	-46
CR_GT_DD005	341930	7621517	1013	850	288	-52
CR_GT_DD006	341655	7621361	1014	750	323	-56
CR_GT_DD007	341314	7621501	1013	801	28	-59
CR_GT_DD008	341221	7621658	1015	786	66	-59
CR_GT_DD009	341297	7622036	1013	450	115	-40
CR_GT_DD010	341545	7622182	1012	900	169	-54
INFRA_GT_DD001	342011	7621291	1013	651	353	-71
INFRA_GT_DD002	341758	7621377	1014	848	310	-67
INFRA_GT_DD003	341561	7621357	1014	1,070	19	-68
INFRA_GT_DD004	341352	7621446	1014	903	34	-69
INFRA_GT_DD005	342103	7621197	1013	600	305	-76
INFRA_GT_DD006	341444	7621168	1015	104	335	-69
INFRA_GT_DD006A	341444	7621168	1015	32	269	-51
INFRA_GT_DD007	341548	7621203	1014	969	9	-55
INFRA_GT_DD008	341985	7621696	1013	1,038	270	-62
INFRA_GT_DD009	341452	7621001	1014	81	350	-69
INFRA_GT_DD010	342174	7621078	1014	60	165	-70
INFRA_GT_DD011	341723	7621092	1013	501	168	-47
INFRA_GT_DD012	341446	7620716	1013	429	346	-64
INFRA_GT_DD013	342036	7621166	1013	519	166	-47
KGR_GT_DD001	341413	7622177	1012	698	157	-52
KGR_GT_DD002	341789	7622069	1012	744	210	-45
KGR_GT_DD003	341974	7621820	1013	897	255	-50
KGR_GT_DD003A	341974	7621819	1012	11	253	-54
KGR_GT_DD004	341907	7621480	1013	849	301	-54
KGR_GT_DD005	341627	7621359	1015	615	346	-61
KGR_GT_DD005A	341559	7621629	515	331	350	-58
KGR_GT_DD006	341324	7621487	1013	711	41	-48
KGR_GT_DD007	341224	7621697	1014	800	87	-43
KGR_GT_DD008	341308	7622047	1013	825	139	-51
KGR_GT_DD009	341683	7622141	1012	636	221	-58

Drillhole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
KGR_GT_DD010	341852	7622008	1012	800	245	-55
KGR_GT_DD011	341614	7621664	869	604	303	-80
Total				**23,958**		

Source: Lucara (2019)

Figure 10-2 shows a cross-sectional view, oriented towards the east, showing the South, Centre and North Lobes shown as green (transparent), red and blue, respectively.

Figure 10-2: Drillholes in the South, Centre and North Lobes (2017-2019)



Source: SRK (2019)

10.2.3 2020 Drilling Shaft Geotechnical Investigation

A shaft geotechnical drilling program took place between May 16 and December 13, 2020. It consisted of two vertical diamond core drillholes designed to trace the proposed UG mine shaft alignments along their entirety. Table 10-4 lists the drillholes with corresponding collar coordinates and final depths. The collar locations and are shown on Figure 10-3. A full geotechnical report on the drill program was prepared by JDS in 2021 (JDS, 2021).

Drillhole VS_GT_DD001 was collared at the approximate center of the proposed V/S and drillhole PS_GT_DD001 was collared at the approximate center of the proposed P/S. Both drillholes were drilled as close to vertical as possible, along the proposed shaft centerlines. Drilling was undertaken by Dewet Drilling Botswana, of Gaborone, using a Buffalo 90 multipurpose drill rig. Coring was completed with an SK 4 ¼“ B core barrel which produces a 120.6 mm diameter hole and 69.0 mm core diameter, capable of drilling up to 6 m core runs.

Drillhole VS_GT_DD001 was drilled with a tricone bit and cased through the upper unconsolidated sediments and calcrete of the Kalahari Fm. Diamond core drilling began recovering weathered basalt at a depth of 18.5 m. Drillhole PS_GT_DD001 was cored from surface with PQ-sized tooling. The first core recovered was at 10.0 m and the PQ-sized coring continued to a depth of 15.5 m. The standard 69.0 mm core size began at 15.5 m and the PQ rods were left in the hole as casing.

Drillhole collar locations were surveyed by the mine survey department. Downhole deviation was surveyed by Poseidon Geophysics of Gaborone using an Axis Champ Navigator gyro tool.

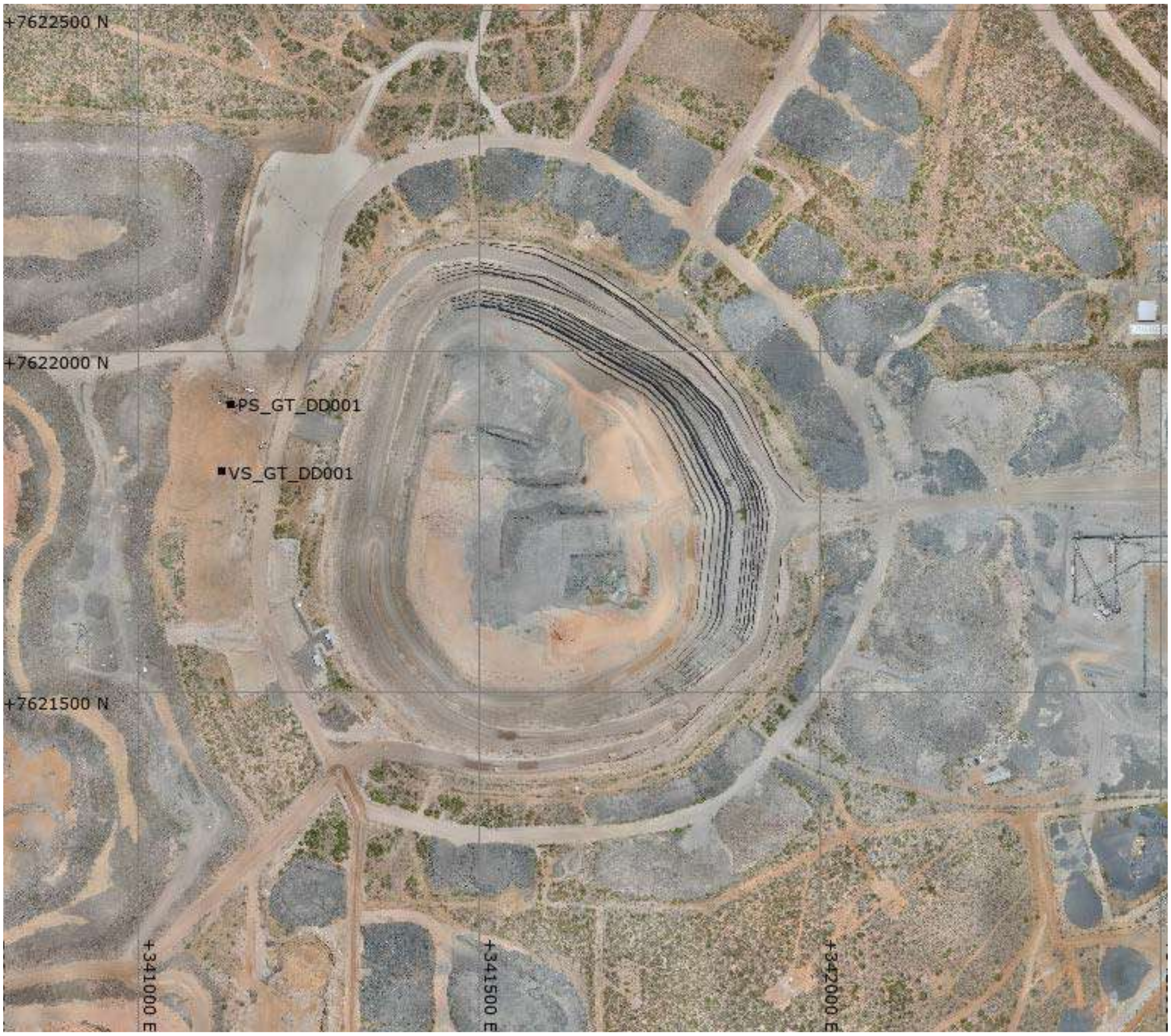
Downhole deviation is discussed in detail in Section 4.1.

Table 10-4: 2020 Shaft Geotechnical Drilling

Drillhole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
VS_GT_DD001	341122.0	7621824.0	1013.0	746.5	-	-90
PS_GT_DD001	341137.0	7621923.0	1012.7	768.0	-	-90
Total				**15,14.5**		

Source: JDS (2023)

Figure 10-3: Geotechnical Drillholes of Proposed Shafts



Source: JDS (2023)

10.3 Drill Core Sampling

Sampling of drill material in support of historical and recent resource estimates is well documented in previous Technical Reports (McGeorge et al., 2010; Lynn et al., 2014; Nowicki et al., 2018). This section summarizes sampling work carried out on the 2018 / 2019 FS program drill cores (Section 10.2) and is restated from Doerksen et al. (2019). A key requirement of the South Lobe Mineral Resource Estimate is the demonstration of geological continuity within the

M/PK(S) and EM/PK(S) units with depth (Sections 7.4.3 and 14.3.5). Sample coverages achieved from the 2018-2019 drill cores in the South Lobe are shown in Figure 10-4. Sampling was undertaken for bulk density, petrography and microdiamond analysis, as follows:

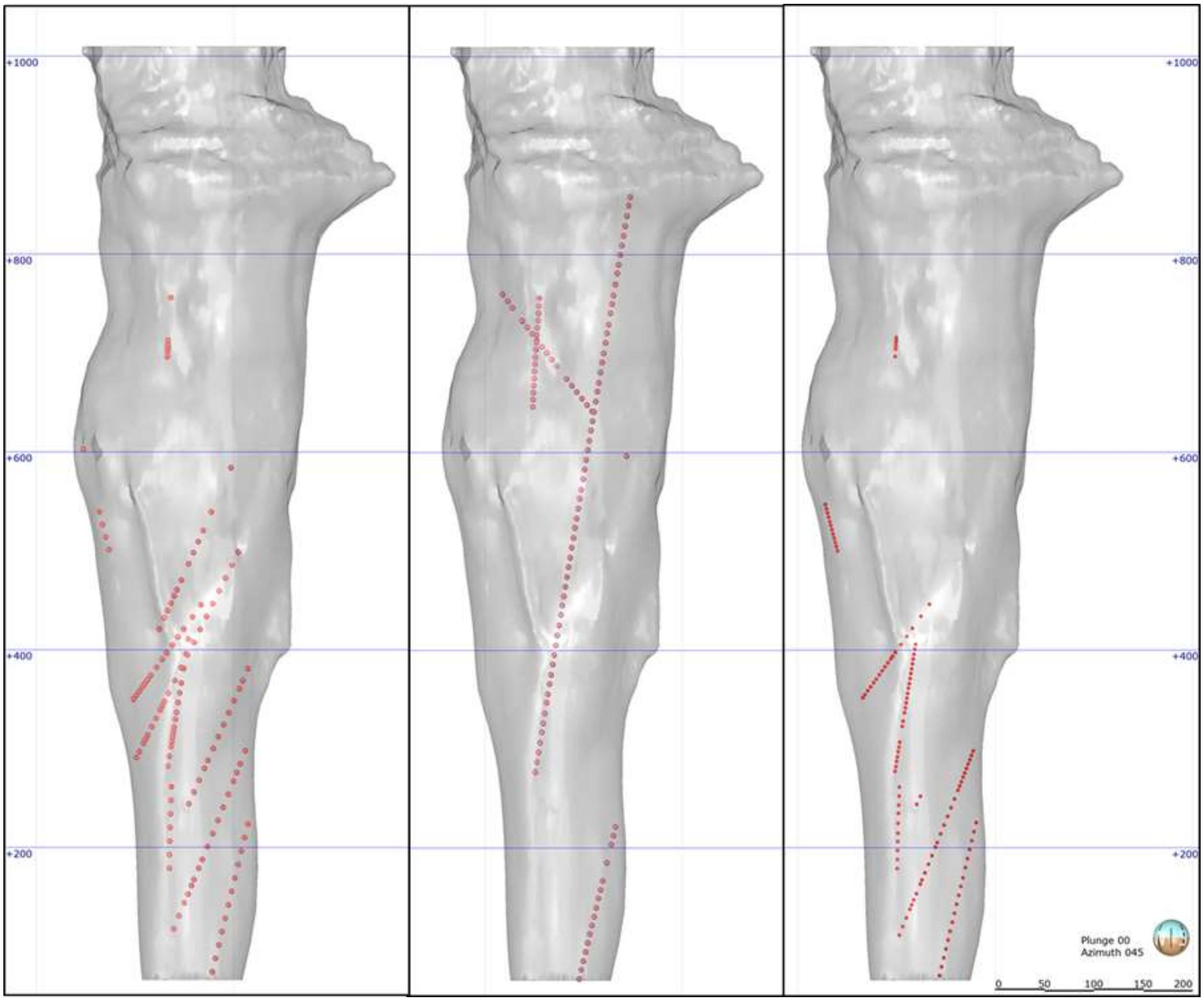
- Bulk density samples (n = 209, of which 188 are in the South Lobe). Samples each comprised 10 cm of whole core and were collected at regular 10 m intervals in six KGR / INFRA drill cores (four of which are in the South Lobe). It is noted that the historical and 2017 drill cores were comprehensively sampled for bulk density. In addition to the bulk density samples in kimberlite, a total of 2,235 bulk density samples (5 to 10 cm length) were collected in country rock in 22 CR-GT / INFRA / KGR holes;

- Petrography samples (n = 128) were collected from 10 of the 14 KGR / INFRA drill cores intersecting the South Lobe, predominantly targeting kimberlite below 450 masl. Samples each comprised 15 to 25 cm of whole core and were collected at regular 10 or 15 m intervals, or in some cases at 5 m intervals, depending on the geology; and

- Microdiamond samples (n = 150) were collected from nine of the 14 KGR / INFRA drill cores intersecting the South Lobe, predominantly targeting kimberlite below 450 masl. 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 between 5, 10 and 15 m depending on the geology and objectives of the sampling.

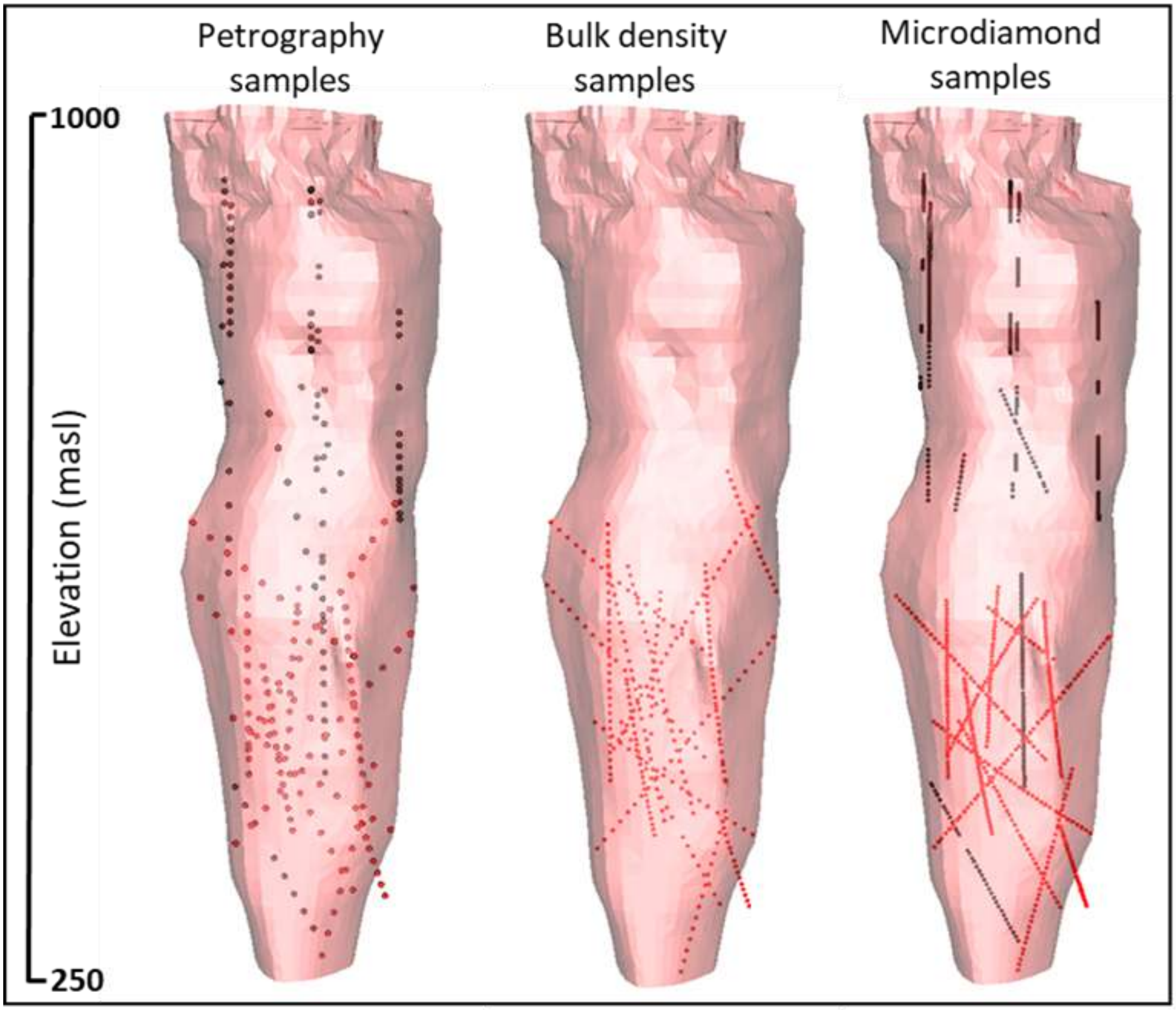
Figure 10-5 shows the locations of samples collected from 2017 and historical drillholes (Nowicki et al., 2018).

Figure 10-4: Location of Samples Collected from 2018 / 2019 Drill Core in the South Lobe



Source: SRK (2023)

Figure 10-5: Location of Samples Collected from Drill Core in the South Lobe during 2017



Source: Nowicki et al. (2018)

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

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 14.3.4.

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 metre 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 samples was passed to the plant foreman. The processing plant was a 10 t/hr 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 Diamond Trading Company (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 practice.

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).

11.2 Petrography Samples

All petrography samples collected in 2017 and 2019 were labelled with the drillhole number, depth and way-up direction by Boteti or Lucara Botswana geologists. No further sample preparation was carried out on site. Petrography samples were shipped to Vancouver Petrographics Ltd. (2017) and Precision Petrographics Ltd. (2019) 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 SRK Consulting office in Vancouver, Canada.

11.3 Bulk Density Samples

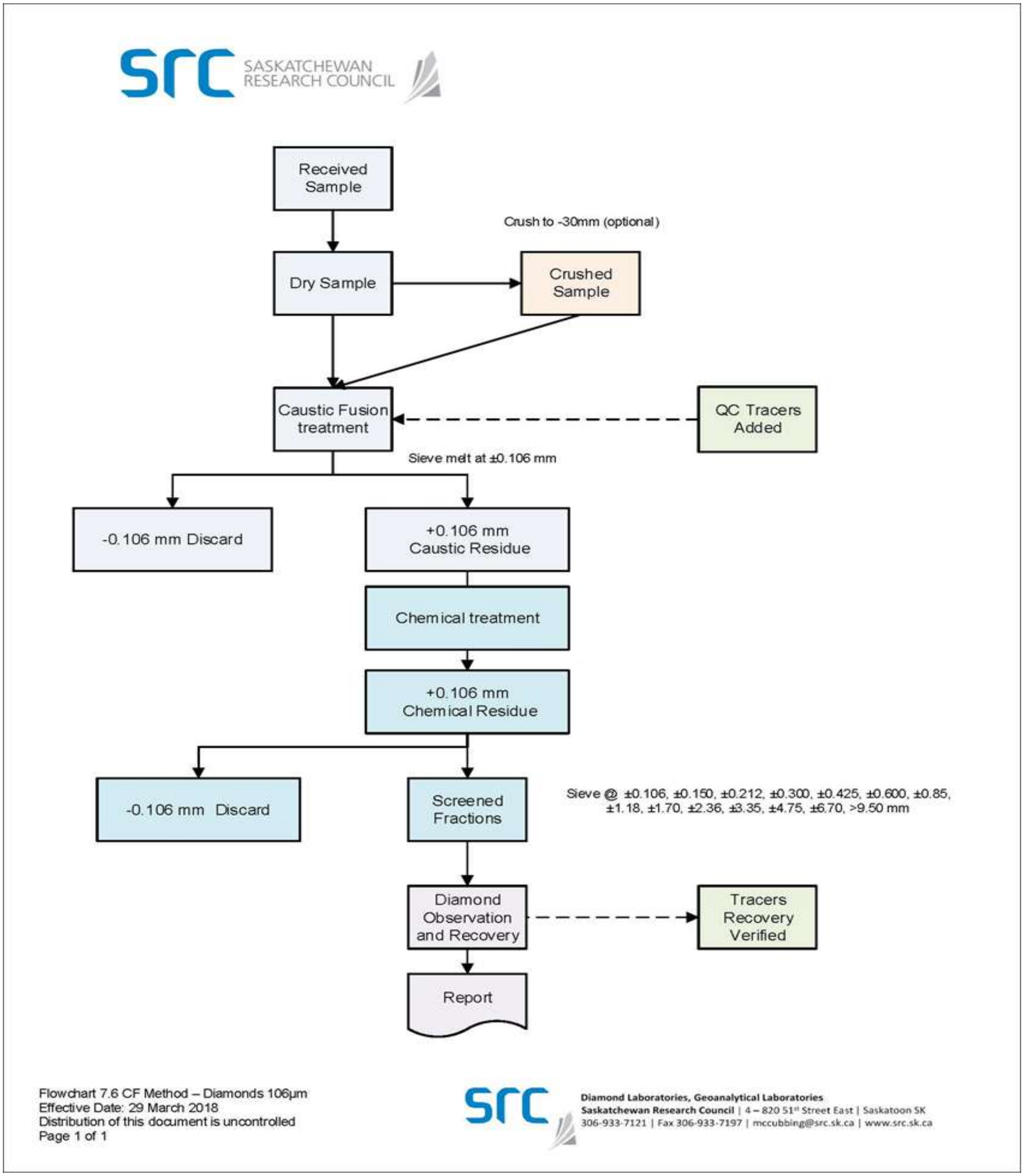
All bulk density sample processing in 2017 was carried out on site by Boteti geologists. Sample masses were recorded at an on-site laboratory and sample volumes were determined by a water-immersion method as per Lipton (2001). No drying of samples was carried out; the bulk density measurements collected 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.

11.4 Microdiamond Samples

No preparation of microdiamond samples collected in 2017 and 2019 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 eight-kilogram 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



Source: SRC (2019)

12 DATA VERIFICATION

12.1 Geological Model

12.1.1 Drillhole Collar and Orientation Surveys

Early (2005-2007) delineation drillholes were surveyed with a Leica DGPS500 system and downhole surveys used magnetic- or gyroscope-based systems, with the magnetic-based surveys considered low confidence (McGeorge et al., 2010). Significant issues with downhole orientation surveys were encountered during core drilling in 2017, such that 11 of 31 pierce points were discarded as unreliable (Nowicki et al., 2018). The 2018/2019 drillholes were surveyed by one or more magnetic-based, inertial, or north-seeking gyroscope tools. Ms. Webb examined the original and reviewed datasets (following comprehensive QA/QC by Lucara) and concluded the data produced by the EZ-Gyro north-seeking tool were the most comprehensive, reliable and suitable for use in the geological model update. Ms. Webb further compared the recent and historical data, and no significant issues or discrepancies were noted.

12.1.2 Geological Logs and Internal Geology

The AK6 geological model is based primarily on drill core logs and petrography (also minor historical whole rock geochemistry). The drillhole database and all core photos were provided to SRK. A comprehensive review and re-logging of historical and 2017 South Lobe drill cores at the mine site and in core photos was undertaken by Ms. Webb of SRK while employed by MSC, resulting in update of the internal geology (re-modelling of the M/PK(S)-EM/PK(S) boundary) as documented in Nowicki et al. (2018) and references therein. Ms. Webb also reviewed all 2018/2019 drill cores intersecting the South Lobe to verify the mine-generated drill logs, and additionally verified the logged contacts in core photos for all holes for which the drill core was not examined.

12.1.3 Internal Dilution Data

Estimates of the volume percent of wall-rock fragments greater than 0.5 cm in size were determined for historical (2005 to 2007) drill core by line scan measurements over 0.3 and 0.5 m intervals at ~4 to 5 m spacing downhole, and for 2017 and 2018/2019 drill core by line scan over 1 m intervals on a continuous basis downhole. The methods are considered by Ms. Webb to be appropriate and consistent with industry best practice, and no inconsistencies between the datasets or between the data and Ms. Webb’s observations of the drill core were noted during a review of the historical and recent data.

After review of the drillhole database, including collar and downhole survey data, geological logs, core photos, and internal dilution estimates, Ms. Webb is of the opinion that the data (excluding the 2017 orientation survey data mentioned above) are sufficiently reliable for use in generation of a geological model of appropriate confidence to support the estimation of Mineral Resources.

12.2 Mineral Resource Estimate

12.2.1 Bulk Density

The bulk density data used for estimation at KDM derives from regular-spaced sampling of historical and recent delineation, pilot and geotechnical drill cores. Mr. Revering considers the methods used to be in line with industry best practice (although notes that details of the procedure used historically are not available). Mr. Revering reviewed the bulk density database, the scale calibration measurements for recent sampling, and verified that samples were correctly coded according to the updated geological model domains. No significant issues or discrepancies were found.

12.2.2 Microdiamond Data

Microdiamond drill core sample results used for Mineral Resource estimation were compiled from original lab certificates. All microdiamond samples were processed at the Saskatchewan Research Council (SRC) in Saskatoon, Canada, which uses a systematic quality control system. Synthetic diamonds (referred to as Tracers) are added to samples prior to caustic fusion and during chemical treatment of caustic residues, and recoveries of these synthetic diamonds are reported along with microdiamond recovery results. Mr. Revering reviewed the microdiamond sample and quality control results, and no significant issues were noted.

12.2.3 Macrodiamond Data

Macrodiamond bulk sample data was obtained from two large diameter sampling campaigns conducted in 2006 and 2007. Mr. Revering compared the macrodiamond bulk sample database to original sampling and process reports and found the data to be consistent with the original bulk sampling documentation.

12.2.4 Production and Sales Data

Production and sales data dating back to the start of mining operations in 2012 were provided to SRK as part of the 2023 Mineral Resource update. Although a detailed audit of this information was not conducted by Mr. Revering, the information was reviewed in the context of reconciling past production and diamond revenues with data used for the 2023 Mineral Resource Estimate. No significant issues or discrepancies were noted by Mr. Revering during this review.

After review of the microdiamond, bulk sample, and production and sales data for KDM, Mr. Revering is of the opinion that the data is sufficiently reliable to use for Mineral Resource estimation.

12.3 Mineral Reserve Estimate

Multiple site visits were conducted by the QP during course of the project.

Mineral reserve estimates were based on the 2019 Mineral Resource block model. The resource block model was imported into the mining software Vulcan and was validated to verify mineral tonnes and grade reported in Section 14.

Surveyed site topography, including OP activities and stockpiles, were used to calculate the Mineral Reserve Estimate. Surveyed data has been received, reviewed, and visually verified on site in periodic intervals since 2019.

Reconciliation efforts performed since 2019 have been reviewed by the QP. Mine Call Factor is a modifying factor used by Lucara which tracks the reconciliation between the block model and actual recovered carats in the process plant. Mine Call Factor is assumed to be 100%, historically this factor has reconciled either near or above 100%, however in the 12-month period prior to the Reserve Statement, the Mine Call Factor has deviated away from historical average performance to approximately 95%. This deviation from historical performance, will require monitoring to ensure this trend is not consistent in future periods.

Cut-off value estimates were based on first-principle cost estimation for the UG reserves and actual costs for OP mining, processing and G&A costs. Operating costs were verified against company financials.

The data and information used to inform the mineral reserve estimate are considered adequate, and representative.

12.4 Mineral Processing and Metallurgical Testing

Eleven buckets containing rocks from the pit and HQ core from UG were shipped to BaseMet Laboratories in Kamloops, B.C. for comminution testwork in 2019. The purpose of the testwork was to determine if the EM/PK(S) and M/PK(S) material was similar throughout the resource with respect to AG milling. The drillholes used for metallurgical testwork were plotted against the planned area to be mined and were found to be spatially representative and provided samples at depth that represent areas of the UG mine. It is the QP's opinion that there is sufficient information and testwork to determine the similarities between the OP and UG EM/PK(S) and M/PK(S) material with respect to AG milling at an FS level.

12.5 Geotechnical

One site visit was conducted by the QP during the course of the project to inspect the rock mass conditions in the P/S and V/S, the OP, and to inspect core.

The QAQC of core logging, borehole (wireline) logging, field testing and sampling activities was carried out by SRK Consulting (South Africa) following standard operating procedures implemented by SRK and was reviewed by the QP.

The P/S and V/S borehole core were analyzed in detail by the QP.

The geotechnical and litho-structural models were reviewed in detail by the QP, in consultation with SRK colleagues who were originally responsible for the work.

The volume of data available for the study is considered adequate. The drilling program included completion of 21,837 m of geotechnical drilling from 35 drillholes through both country rock and orebody to support 7,385 field strength (point load) tests and a broad spectrum of laboratory tests encompassing 3,501 total samples. The Total Level of Data Confidence (TLDC) was quantified specifically for the laboratory testing specimens and indicates that the majority of tests met the minimum criteria for the upper limit of the feasibility level study of between 60 - 75%. Lower levels of confidence were obtained for specific thin subdomains within the Tlapana formation and is related to the small volume of materials available for sampling.

12.6 Water Management and Hydrogeology

KDM is a brownfields site with eleven years of actual mine dewatering data available (2012-2019) on which the aquifer system behavior and pressure response could be analyzed and used in the model calibration. The subcomponents that fed information to the LOM dewatering strategy and design consist of 27 specialist reports. The level of data gathered and analyzed is beyond feasibility study requirements with 23 pumping tests, 58 packer tests and 400 hydrochemistry tests. Existing data was reviewed and analyzed statistically for quality assurance.

- The data gathering was completed or overseen by suitably qualified personnel and reviewed by senior project specialists;
- Data verification was completed by statistical analyses for spatial and temporal data sets;
- Aquifer tests were checked against standard procedures for constant discharge and recovery tests done in the pre-operational phase and packer tests done during the feasibility study;
- Hydrochemical and geochemical tests were completed at accredited laboratories;
- Limitations in data sets were listed and clear recommendations were made to address the gaps; and
- Limitations were conservatively accommodated in the modelling and decision-making process so that impacts are over- rather than under-estimated in terms of risks and costs, in line with the precautionary principle.

The level of data available is adequate and even beyond FS requirements.

12.7 Mining Methods

Multiple site visits were conducted by the QP during the course of the project to enact the following data verification procedures:

- On-site Meetings with Technical and Operational staff along with a review of relevant site reports and studies;
- Inspection of core shack, logging practices, and borehole collars to assist in geotechnical verification procedures referenced in Section 12.5;
- Inspection of proposed UG entry (shaft) locations to verify offset distances from OP and other existing and planned mine infrastructure;
- Inspection of site facilities such as workshops, camps, offices, explosives manufacturing and storage, and laydowns to verify areas which can support UG development and those which require expansion;
- Review of blast fragmentation as observed in pit and as stated in blast reports to verify blasting parameters for use in UG production stoping;
- Import and validation of resource block models to verify mineral tonnes and grade reported in Section 14; and
- Construction oversite of shafts and shaft sinking infrastructure to qualify capital estimates and timelines.

It is the QP's opinion that there is sufficient data in quantity and quality for the purposes used in the technical report.

12.8 Environmental Studies and Permitting

The data and information relating to environmental and social aspects of the Project were KDM's (a) Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) for the KDM OP mine (on-going operation) and UGP, (c) data and studies provided by Lucara, (d) environmental and social monitoring reports relating to the on-going construction of the KDM UGP by the Independent Environmental and Social Consultants (IESC), and independent third-party assured 2022 sustainability report, as cited in the section "Environmental Studies" and "References".

The information provided in this report is provided without limitations. The qualified person has over 25 years of relevant experience. His most recent visits to KDM were from 27-28 April 2021, 24-25 November 2022, and 13-17 February 2023. Based upon this, the qualified person is confident that the information provided is adequate for the purposes used in the technical report.

12.9 Process Description / Recovery Methods

The following steps were taken as qualified person to verify the data reported in Section 17 of the KDM UG Feasibility Study Technical Report:

- To successfully assess current plant performance and production, a site visit was conducted on September 2 and 3, 2019 at KDM, Letlhakane, Central Botswana. During the site visit Lucara Botswana and Lazenby employees (contract operators responsible for the running and maintenance of the processing operations) were engaged and consulted to source the desired information and data as part of the overall treatment plant evaluation;

- The Process Design Criteria (PDC) tabulated values were verified (reviewed, approved and signed-off) by the client during the Phase I and II implementation of the respective KDM projects. The overall KDM Block Flow Diagram (BFD) was also verified through previous project engagement(s)/verifications and subsequently amended post site visit early September ’19 to confirm recent changes/upgrades. The List of Major Components (summary Mechanical Equipment List for Installed Drives ≥ 100 kW) was verified (reviewed, approved and signed-off) by the client during all implementation phases of the respective KDM projects. The 2018 Plant Performance, Treatment Plant Key Feed Stream PSDs, Raw/Total Water Consumption and Energy Consumption figures were actual information sourced from site; converted into graphical representations for ease of reference, interpretation and reading. The Key Screen Panel Aperture Summary and Crusher Closed Side Setting (CSS) tabulated data were also actual operational information obtained from and confirmed by Lucara Botswana; and

- No limitations and/or failure to conduct such verification were encountered.

It is this qualified person’s opinion that the data utilized and represented is adequate and compliant for the purposes used in the technical report – with specific reference made to Section 17 of this report.

12.10 Project Infrastructure and Services

12.10.1 Power Supply

The QP reviewed power supply invoices from Botswana Power Corporation to verify power supply prices and consumption of existing facilities.

The QP prepared life of mine peak power demand estimates based on equipment lists and project schedules to inform the design of power supply infrastructure.

The QP witnessed site power supply upgrades made between 2020 and 2023 which includes:

- New temporary diesel genset farm with 15 MW capacity;

- New 132kV off-site substation expansion at the Letlhakane substation;

- New overhead power line from Letlhakane substation to KDM minesite;
- New on-site substation and 11 kV distribution;
- New overhead and trenched 11 kV power line from on-site substation to sinking terrace;
- New UGP 11 kV E-house and switchgear; and
- New permanent diesel genset farm with 6 MW capacity.

The QP is of the opinion that the power supply is sufficient for the intended life of mine plans.

12.10.2 Roads, Buildings, and Facilities

The QP visited site buildings and facilities on several occasions to verify size, condition, suitability for use, and to identify expansion requirements. Infrastructure inspected includes:

- Off-site access road from Letlhakane, and on-site access roads between offices, OP, and UG pads;
- 200-person camp including dormitories, kitchen, laundry, gym, security, fencing;
- Site Security fencing and access controls;
- Administration and OP Engineering Offices;
- Medical Clinic, Fire truck, Ambulance, and emergency mobile winder;
- Core Shed and Exploration Buildings;
- OP, primary tip, rock breaker, and jaw crusher;
- Stormwater Management Pond(s);
- Explosive Magazines and Bulk Emulsion Storage;
- UG Terraces, Shafts, Offices, Workshops, Laydowns, and Warehouses;
- UG service lines including Filtered Water, Service Water, Sewerage, Dewatering, Compressed air, and power;
- Storage facilities and distribution infrastructure for major consumables including fuel, cement, and aggregate; and
- Waste Rock Dumps.

The QP is of the opinion that the project infrastructure is in good working order and well suited for the life of mine requirements, pending UGP specific installations yet to come.

12.11 Residue Storage Facilities

Knight Piésold visited the mine site on a number of occasions to meet site personnel to obtain production data, operating details, conduct site inspections of the FRD and CRD, and to undertake geotechnical investigations. Laboratory testing was done on in-situ soils, construction materials, slimes and tailings samples. A design criteria was compiled and approved. By means of an internal review process, the QPs are satisfied that the level of information is fit and appropriate for the feasibility design work that has been completed. Drawings have been produced on which bills of quantities have been compiled. The cost estimate for the FRD and CRD facilities is therefore deemed realistic for both capital, work capital and operating costs for the planned life of mine, and the associated construction schedule for wall raising and conveyor extensions. The information is adequate for a feasibility study.

12.12 Capital and Operating Cost Estimates

Lucara Botswana has provided for capital and operating costs for OP mining, processing, tails, G&A, cost of sales, taxes, financing, and mine closure. Lucara Botswana costs have been validated against annual financial reports.

All other project capital and operating costs have been estimated or managed by JDS through a blend of contractor quotes and first principal estimates using actual regional consumable costs, contractor costs and labour rates. Multiple bids have been used to validate consumable, equipment, and contractor costs. Mine Operating costs have been benchmarked against operations of similar size and mining method.

The information used to generate the capital and operating costs is adequate for a feasibility study. Cost estimates are considered AACE Class 3 with a 40% maturity of project definition.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineral Processing Testwork

The KDM processing plant has been treating unweathered South Lobe ore since 2015 and mineral processing characteristics are very well understood. For this FS, however, it was deemed appropriate to conduct two confirmatory tests to verify the compatibility of the ore at depth in the current processing plant.

A comminution test program was conducted to test the milling characteristics of the South Lobe material below the OP to determine if the mill is suitable for deeper EM/PK(S) ore.

The second test involved testing of Tomra's X-ray Transmission (XRT) machines and associated software to determine their ability to differentiate between diamonds, coal, carbonaceous shale and other waste rock. Due to the high carbon content of coal and carbonaceous shales, they were of greatest concern. The dilution of ore with carbonaceous shales (and the small, sporadic, coal seams contained therein) is anticipated to occur during the later stages of mine life. Testing was conducted by Tomra at their testing facilities in Germany.

13.2 XRT Testwork

Various drill core samples from the 2019 FS drilling program were collected and prepared from representative areas of the planned UG mine. The core was cut into discs of 2 to 30 mm in thickness and shipped to Tomra's lab for testing with their COM Tertiary XRT unit. (See Figure 13-1 for samples).

Figure 13-1: Ore and Waste Samples Prepared for XRT Testing

Source: Tomra Sorting (2019)

The COM Tertiary XRT can distinguish between liberated diamonds and different host rock lithologies. The sensor images show that all the waste lithologies provided can be correctly recognized by the sensor, thus, the XRT technology is applicable for the wider range of lithologies encountered in UG operations. The results of the First Inspection Report (Tomra 2019) showed that the carbonaceous mudstone can be recognized by the XRT as waste by using a standard setting.

In spite of the positive test results, the exclusion of dilution from all types of waste rock, and particularly carbonaceous shale will be an important factor in UG mining, and the mining method has been planned accordingly.

13.3 Comminution Testwork

Bulk and HQ drill core representing EM/PK(S) and M/PK(S) zones of the deposit were selected by the site representatives and shipped to Base Metallurgical Laboratory (BaseMet) in Kamloops, BC in 2019. Eleven samples in total were received, which included bulk rock samples and drill core from both zones at varying depths. Several comminution tests on both the bulk and variability samples were completed. The results demonstrated that the two zones, EM/PK(S) and M/PK(S), are similar in hardness with respect to the bulk and variability samples (Doll 2019 and BaseMet 2019).

13.3.1 Sampling

A list of the samples received, and the location of the samples are shown in Table 13-1, Figure 13-2 and Figure 13-3.

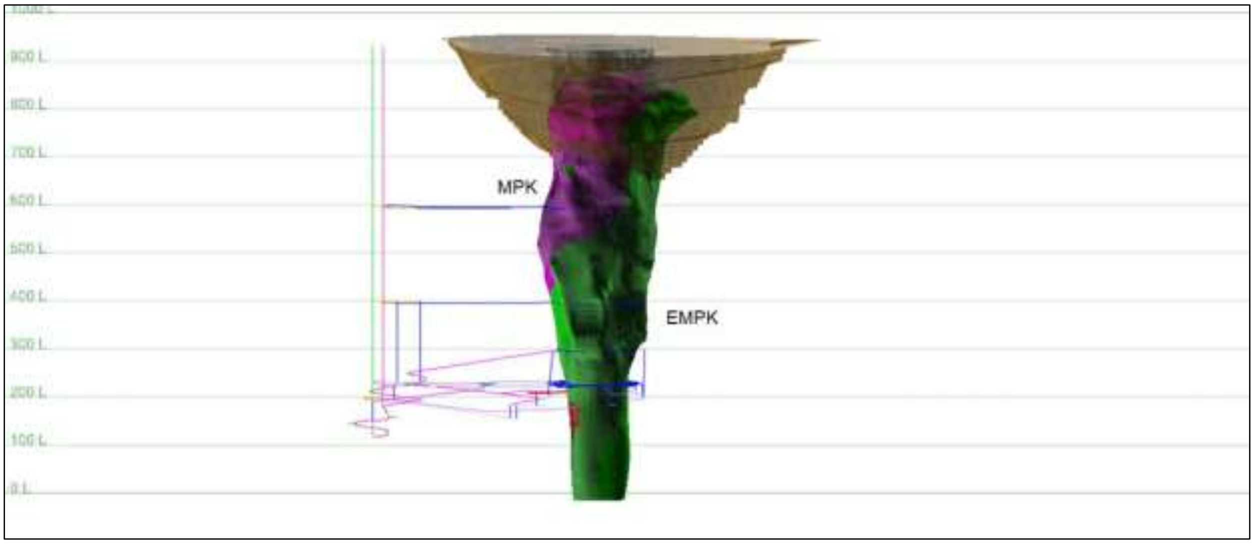
Table 13-1: Comminution Testwork Sample Selection

Sample ID	Hole ID	From (m)	To (m)	Lithology*	From	Mass (kg)
KGR_GT_DD002_COM01	KGR_GT_DD002	550	560	KIMB2	Full core (HQ)	29.82
KGR_GT_DD004_COM01	KGR_GT_DD004	774	786	KIMB3	Full core (HQ)	30.00
KGR_GT_DD006_COM01	KGR_GT_DD006	545	555	KIMB2	Full core (HQ)	29.90
KGR_GT_DD007_COM01	KGR_GT_DD007	600	610	KIMB4	Full core (HQ)	29.94
KGR_GT_DD008_COM01	KGR_GT_DD008	755	765	KIMB4	Full core (HQ)	30.06
KGR_GT_DD011_COM01	KGR_GT_DD011	260	270	KIMB2	Full core (HQ)	30.04
KGR_GT_DD011_COM02	KGR_GT_DD011	475	490	KIMB4	Full core (HQ)	29.92
EM/PK(S) (8)	-	-	-	-	Bulk Rock	50.32
EM/PK(S) (9)	-	-	-	-	Bulk Rock	50.46
M/PK(S) (10)	-	-	-	-	Bulk Rock	50.04
M/PK(S) (11)	-	-	-	-	Bulk Rock	50.00

Notes:
*KIMB3/4 represents EM/PK(S) and KIMB2 M/PK(S)

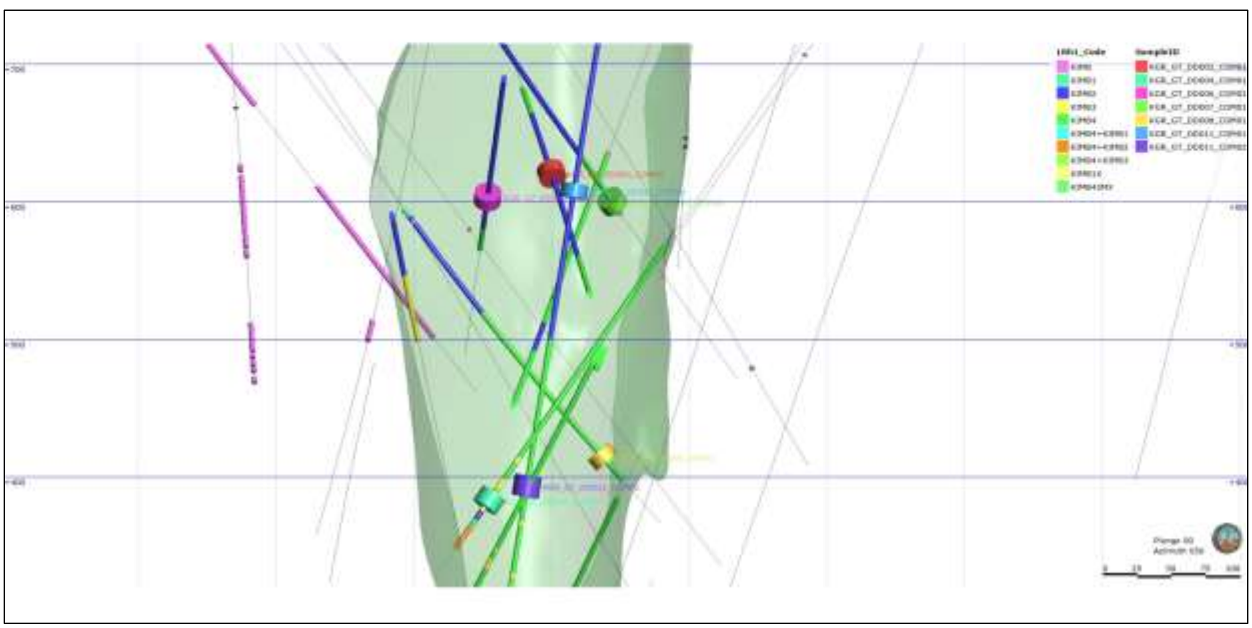
Source: JDS (2019)

Figure 13-2: M/PK(S) and EM/PK(S) Zones



Source: JDS (2019)

Figure 13-3: Drillhole Sample Locations



Source: JDS (2019)

13.3.2 Bulk Sample Test Results

Bond crushing work index (CWi), Bond Rod Mill Work index (RWi), Bond Ball Mill Work index (BWi), and JK Drop Weight tests were completed on the bulk EM/PK(S) and M/PK(S) samples. The results demonstrated that M/PK(S) material was harder with a CWi of 17.0 kWh/t compared to EM/PK(S) with a CWi of 14.2 kWh/t. The RWi results were 18.9 kWh/t and 16.8 kWh/t for EM/PK(S) and M/PK(S), respectively. The BWi at grind sizes of 300, 212, and 150 µm ranged between 23.7 kWh/t and 25.1 kWh/t. Both samples would be classified as very hard at these size fractions. The JK Drop Weight testwork indicates that the material would be considered moderately hard with Axb values of 38.0 for EM/PK(S) and 43.5 for M/PK(S). The bulk sample test results are listed in Table 13-2.

Table 13-2: Summary of Bulk Sample Comminution Test Results

Sample ID	Axb	SG	ta	SCSE	CWi (kWh/t)	RWi (kWh/t)	CSS (µm)	BWi (kWh/t)
EM/PK(S)	37.96	2.96	0.31	10.8	14.2	18.9	300	24.2
EM/PK(S)							212	25.1
EM/PK(S)							150	24.7
M/PK(S)	43.54	2.88	0.30	9.88	17.0	16.8	300	25.1
M/PK(S)							212	24.1
M/PK(S)							150	23.7

Source: BaseMet (2019)

13.3.3 Variability Testwork

Continuous intervals of drill core representing EM/PK(S) and MP/K(S) at different elevations in the ore body were collected and composited to create seven different variability composites. The results indicate the SAG Mill Comminution (SMC) and BWi are similar for all samples tested. The RWi ranged from 17.3 kWh/t to 21.5 kWh/t and the BWi from 22.1 kWh/t to 25.8 kWh/t. The samples would be considered hard to very hard. The variability composites tested with M/PK(S) being slightly softer and did not demonstrate a significant correlation between hardness and depth. The summary of the variability testwork is outlined in Table 13-3.

Table 13-3: Summary of Variability Samples Comminution Testwork

Sample ID	Ore Zone	DWi	DWi	Mia	Mih	Mic	A	b	Axb	SG	ta	SCSE	F_{80} µm	P_{80} µm	Gpr	RWi	CSS µm	F_{80} µm	P_{80} µm	Gpr	BWi
		kWh/m^3	%	kWh/t	kWh/t	kWh/t										kWh/t					kWh/t
KGR_GT_DD002_COM01	M/PK(S)	8.94	78	21.7	17.0	8.8	74.2	0.46	34.1	3.05	0.29	11.6	7772	935	6.74	19.1	300	2794	188	0.98	25.0
KGR_GT_DD004_COM01	EM/PK(S)	7.60	62	20.9	15.9	8.2	74.9	0.49	36.7	2.78	0.34	10.5	8950	970	6.27	19.8	300	2397	202	1.03	25.8
KGR_GT_DD006_COM01	M/PK(S)	9.20	80	22.3	17.6	9.1	83.8	0.39	32.7	3.04	0.28	11.8	8702	864	6.83	17.3	300	2586	215	1.18	23.8
KGR_GT_DD007_COM01	EM/PK(S)	8.31	71	21.5	16.6	8.6	75.8	0.46	34.9	2.90	0.31	11.1	7491	914	7.14	18.2	300	2542	202	1.23	22.1
KGR_GT_DD008_COM01	EM/PK(S)	8.26	71	21.6	16.7	8.6	68.2	0.51	34.8	2.87	0.31	11.0	9571	998	5.53	21.5	300	2739	182	0.98	24.4
KGR_GT_DD011_COM01	M/PK(S)	8.29	71	20.4	15.8	8.2	74.9	0.49	36.7	3.05	0.31	11.1	8581	925	6.77	18.4	300	2513	202	1.05	25.1
KGR_GT_DD011_COM02	EM/PK(S)	9.30	81	22.3	17.6	9.1	79.9	0.41	32.8	3.06	0.28	11.8	9357	907	6.01	19.1	300	2622	184	0.99	24.6

Notes:
*Size Fraction Tested -31.5+26.5 mm

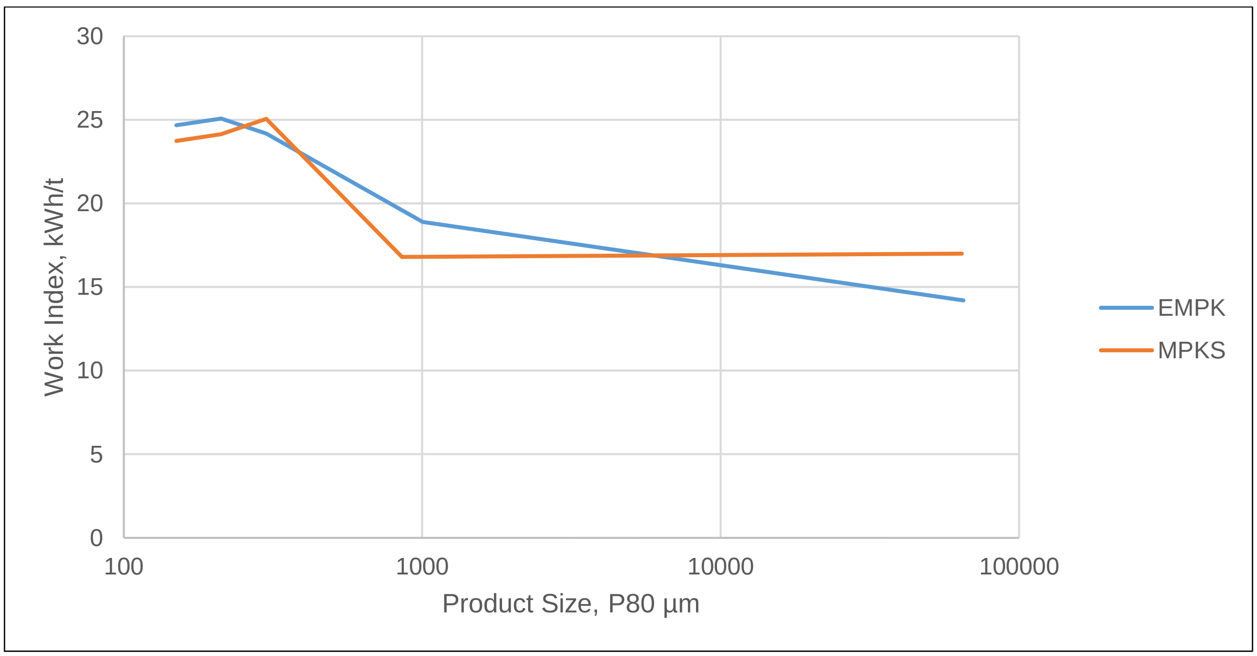
Source: BaseMet (2019)

13.3.4 Technical Evaluation of the EM/PK(S) and M/PK(S) Zones with Respect to AG Mill Operation

The comminution results from BaseMet were compiled and evaluated by Alex G. Doll Consulting Ltd. (AGD) in 2019 to determine if the future material planned to be mined is different from the current material being treated in the existing AG Mill. A review of the samples tested demonstrated that there was not a significant difference between the pit bottom composite samples and the drill core variability samples. The samples tested are amenable to milling in the existing AG process plant.

Figure 13-4 illustrates the work index (kWh/t) as a function of particle size (80% passing, P_{80} µm). The results for the EM/PK(S) and M/PK(S) suggest that both samples are more competent at a finer particle size and have similar curves.

Figure 13-4: Work Index versus Product Size



Source: JDS (2019)

In addition to the comparison of the EM/PK(S) to the M/PK(S) material, the results were graphed against the AGD global database and historical results from other programs. The following observations were made:

- RWi vs. BWi demonstrated that the two samples are very similar and were amongst the hardest samples in the AGD global database. It was noted that historical results did not fit with the recent tests completed by BaseMet or the AGD global database;

- Drop Weight Axb vs. BWi showed minor differences between the drill core and bulk samples. The differences are due to apparatus and are therefore not significant. The BWi for the samples indicated the material is very hard but the Axb value shows the samples to be slightly softer compared to the AGD global database;

- The RWi vs. CWi shows all the samples to be in the hard range and similar to one another;

- Drop Weight Axb vs. CWi showed a minor difference in hardness between the bulk samples and the drill core due to the testing procedure using full JK Drop Weight vs. SMC test. The difference here is not considered significant;

- BWi vs. Product Size P_{80} showed there was little variation in BWi kWh/t at the size fractions tested (300, 212, and 150). No significant difference was observed between the bulk and variability samples; and

- No significant difference between the bulk and variability samples was noted when comparing BWi in g/rev vs Product Size or Ore density vs. BWi in kWh/t.

13.4 Processing Assumptions

The current actual processing recoveries have been used within the Mineral Resource Estimate to determine recoverable grades model curves for the KDM ore.

The KDM processing plant was assumed to support an annual throughput of 2.7 Mt of feed.

14 MINERAL RESOURCE ESTIMATE

The KDM has been in operation since 2012, and as of the end of June 2023, the mined OP extends to a depth of approximately 226 m below surface. The June 2023 Mineral Resource update for the KDM is predicated on the following information:

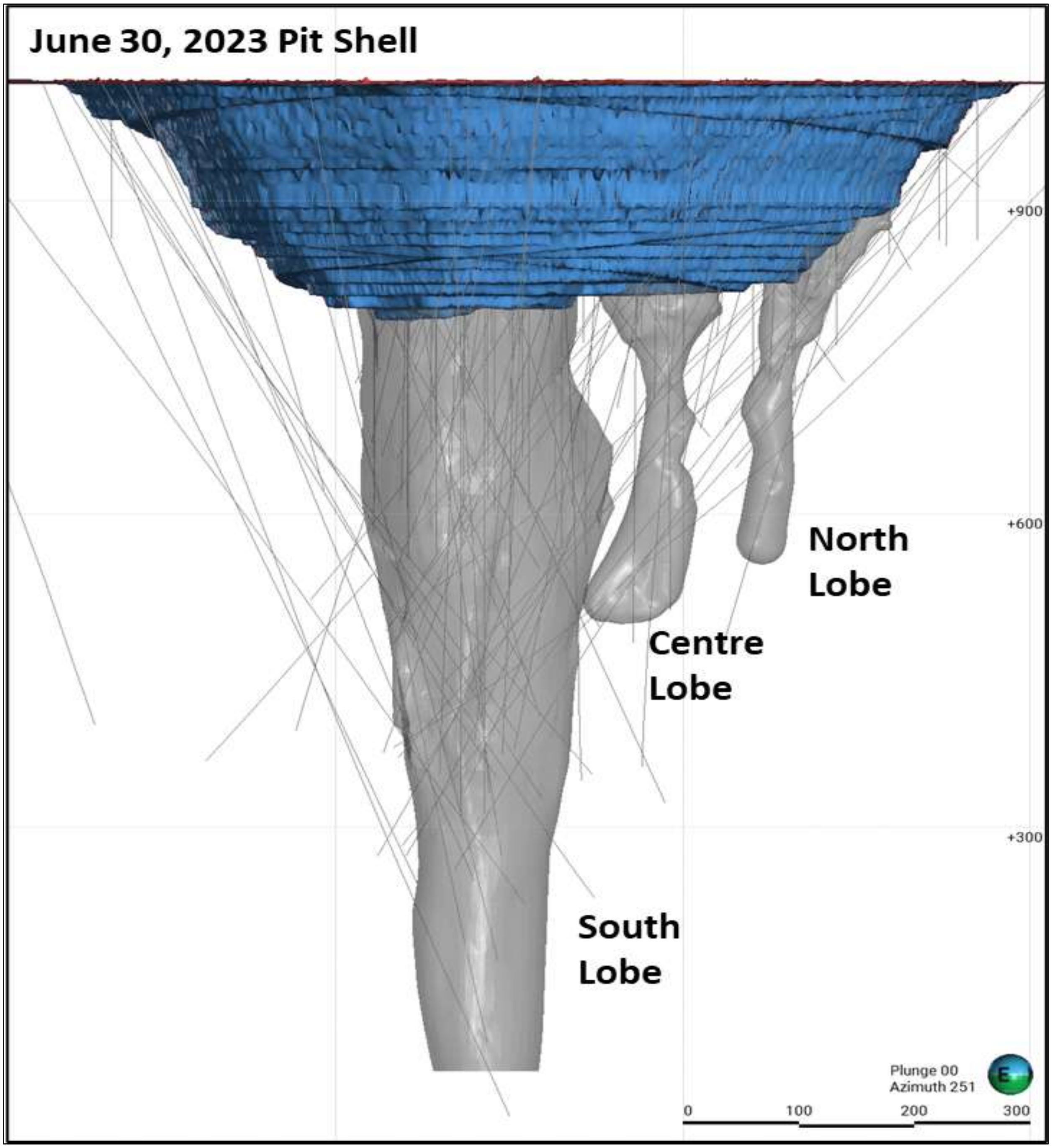
- Diamond core drilling conducted in 2018 and 2019 (located mainly below 600 masl within the South Lobe including a deep extension);
- A geological model for the South Lobe incorporating 2018 and 2019 drilling information;
- Microdiamond sampling of 2018 / 2019 drillholes (specifically targeting internal kimberlite domains within the South Lobe);
- In-pit mapping data of external kimberlite contacts within North, Centre and South Lobes;
- Updated size frequency distributions (SFD) and revised diamond pricing information based on production and sales data to the end of June 2023; and
- As-built survey of the OP mine as of June 30, 2023.

The terms microdiamond and macrodiamond within the context of this report are defined as follows:

- Microdiamonds:
  - Diamonds typically smaller than 0.85 mm that have been recovered from kimberlite drill core using caustic fusion, and a bottom screen size of 105 µm (0.105 mm).
- Macrodiamonds:
  - Diamonds recovered from bulk samples or mine production through conventional crushing of kimberlite ore and commercial diamond recovery techniques. These diamonds are typically larger than 1.00 mm in size, however the recovery efficiency of small diamonds is dependent on the configuration of the process plant and targeted bottom size cut-off.

Figure 14-1 shows the geological model of the kimberlite, the mined OP as of June 30, 2023, and all drilling used to support the current Mineral Resource Estimate for the KDM.

Figure 14-1: Geological Model of the KDM Kimberlite



Note:
Kimberlite pictured in (grey), the June 30, 2023 mined OP, and all drillhole traces.

Source: SRK (2023)

The current geological model and Mineral Resource Estimate were conducted in Seequent's Leapfrog Geo modelling software. The block model is comprised of a sub-block format using the following configuration parameters:

- Block model X, Y, Z origin of 342198, 7622304, 1090, respectively, with no rotation;
- Parent block size of 12 x 12 x 12 m, and a sub-block size of 3 x 3 x 3 m; and
- Model extents (by # of parent blocks) of 109, 92 and 88 along the X, Y, Z axes.

The block model contains local estimates of volume, density and tonnes for all lobes and internal geological domains, and local estimates of diamond grade for the North and Centre Lobes, and the South Lobe M/PK(S) and EM/PK(S) internal domains above 604 and 568 masl, respectively. Global grades are estimated for all remaining volumes of South Lobe M/PK(S), EM/PK(S) and KIMB3 internal domains. Further details of the estimation methodology are provided in the following sections.

14.1 Resource Domains and Volumes

The internal geological model for KDM is described in Section 7.3 of this report, and volume estimates of the unmined, in-situ internal kimberlite domains are listed in Table 14-1. All internal domains that have been mined as of June 30, 2023, are excluded from the volume estimates provided in Table 14-1.

Table 14-1: In-situ Volumes of Unmined Kimberlite Domains as of June 30, 2023

Kimberlite Domain	Volume (Million m^3)	Volume (% of total)
South_M/PK(S)	7.12	42.7
South_EM/PK(S)	8.18	49.0
South_KIMB3	0.32	1.9
Centre	0.68	4.1
North	0.38	2.3
TOTAL	**16.67**	**100**

Source: SRK (2023)

14.2 Bulk Density

A total of 2,796 dry bulk density measurements have been collected from drill core within the kimberlite, of which 2,316 are located below elevation 950 masl which approximately corresponds to the lower boundary of the upper calcretized and weathered kimberlite and country rock breccia zone. Average dry density values within this upper zone in all three lobes are significantly lower than density values below this weathered horizon and therefore have been excluded from the summary statistics provided in Table 14-2. Figure 14-2 provides a colour-coded dry density (units of g/cm^3) sample location map, depicting the base of the upper weathered zone at approximately 950 masl elevation.

Additional dry density sample details for the two dominant kimberlite domains in the South Lobe (i.e., M/PK(S) and EM/PK(S)) are provided in Figure 14-3. As can be seen in the depth profiles for both the EM/PK(S) and M/PK(S) domains a relatively consistent dry density of 2.9 to 3.1 g/cm^3 is observed below a depth of approximately 450 m below surface (560 masl), which roughly corresponds with the base of the Tlapana Shale country rock unit and top of the granite basement. Above this depth horizon, lower dry density values are observed predominately along the margin of the pipe and are considered to be associated with weathering / alteration of the kimberlite along the country rock contact. This is particularly noticeable within the EM/PK(S) density data and is likely due to this unit being constrained to a narrow zone along the eastern margin of the South Lobe above the 450 m depth (refer to Figure 14-4).

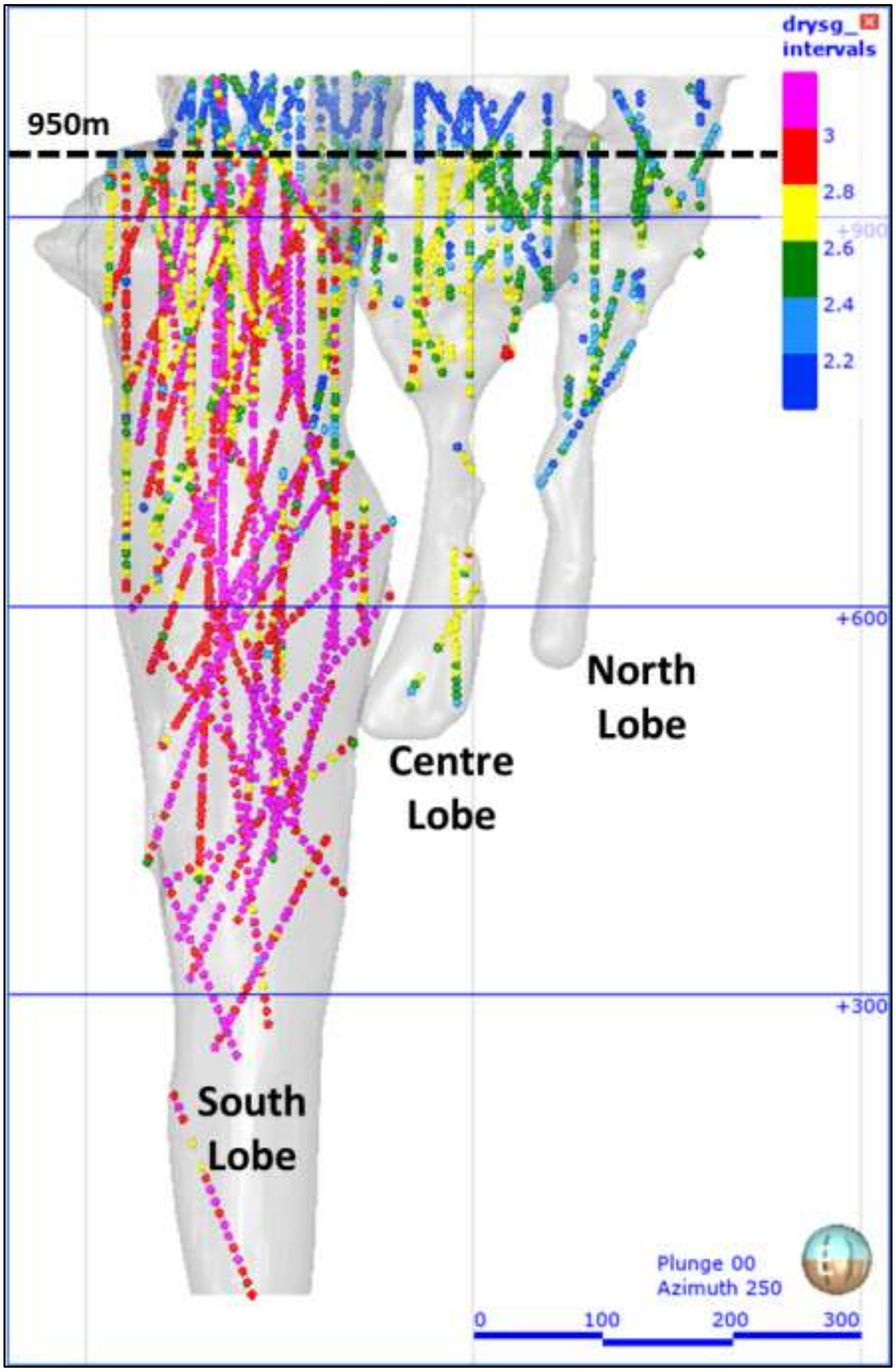
Table 14-2: Average Dry Bulk Density Sample Statistics for KDM Kimberlite Domains

Kimberlite Domain	Sample Count	Mean (g/cm^3)	Standard Deviation (g/cm^3)	Coefficient of Variation	Min (g/cm^3)	Median (g/cm^3)	Max (g/cm^3)
South_M/PK(S)	1,237	2.93	0.19	0.07	1.81	3.00	3.23
South_EM/PK(S)	541	2.87	0.18	0.06	2.07	2.91	3.22
South_KIMB3	14	2.78	0.28	0.10	2.31	2.81	3.08
Centre	370	2.59	0.17	0.06	1.93	2.62	2.95
North	156	2.42	0.16	0.07	1.85	2.45	2.76

Note:
Below 950 masl

Source: SRK (2023)

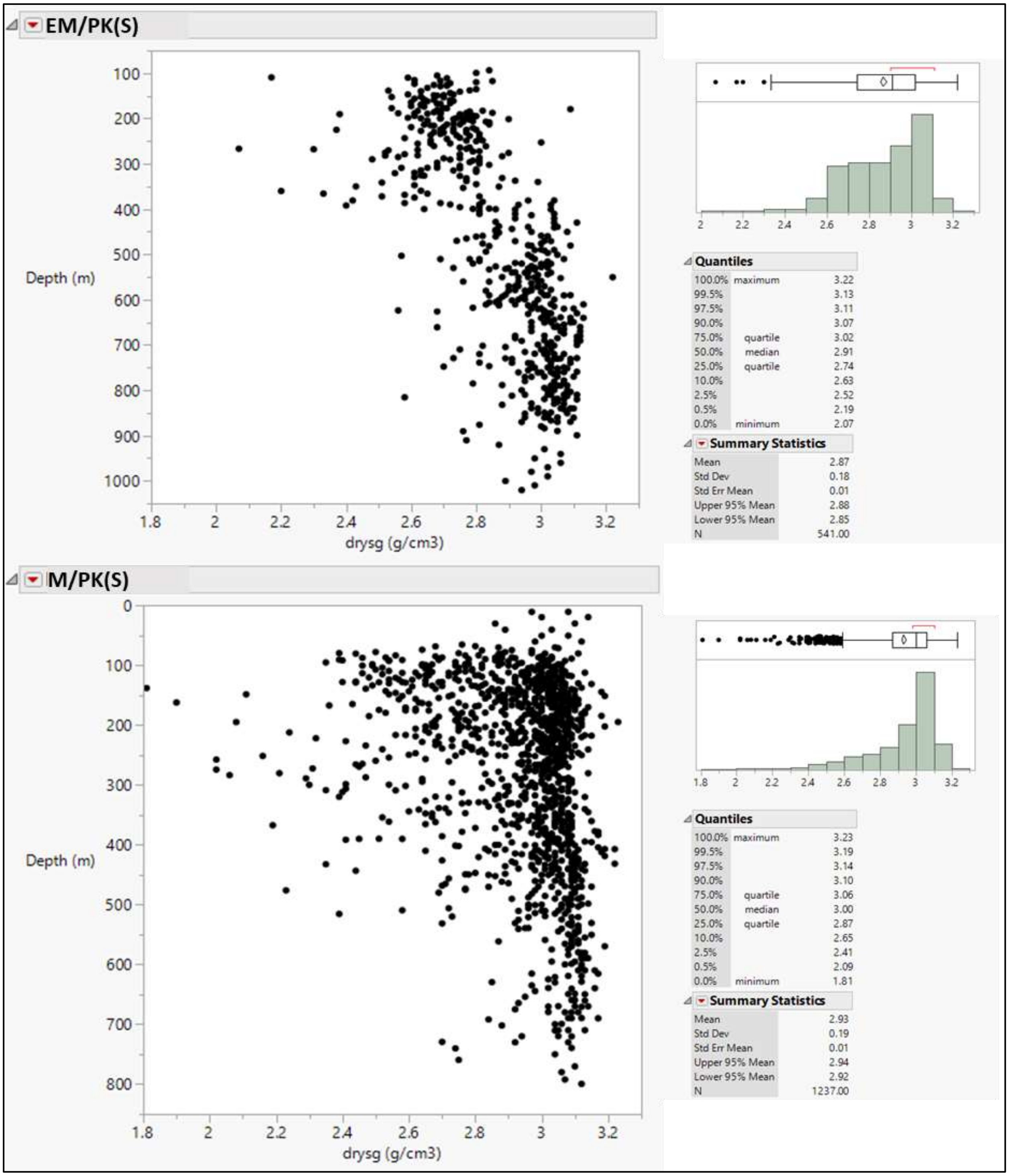
Figure 14-2: Drill Core Dry Bulk Density Sample Location Map



Note:
Dry density units of g/m^3. Black dashed line at 950 masl demarcates approximate extent of upper weathered zone reflected in generally lower densities.

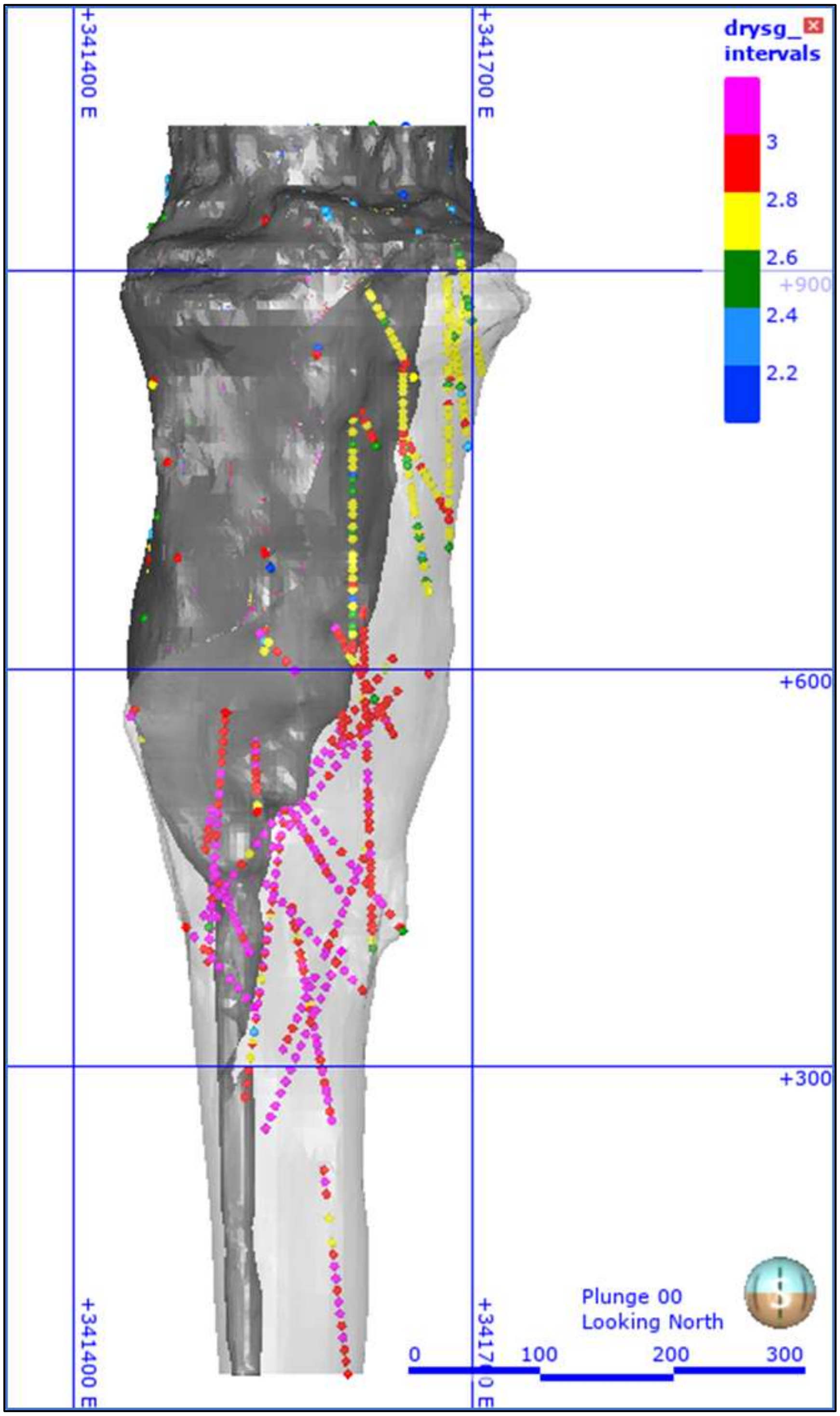
Source: SRK (2023)

Figure 14-3: Dry Density Sample Details for South Lobe M/PK(S) and EM/PK(S) Domains



Source: SRK (2023)

Figure 14-4: South Lobe EM/PK(S) Dry Density Profile with Depth



Source: SRK (2023)

14.2.1 Bulk Density Estimation

Block model estimation of dry density was conducted on a kimberlite domain basis, using hard boundaries between domains to isolate sample populations. The one exception to this was for the South Lobe KIMB3 domain, where a soft boundary was used due to limited available sample data for KIMB3. A "hard boundary" implies that only samples located within a kimberlite domain are used for estimation within that domain, whereas a "soft boundary" allows samples located outside of a domain (i.e., from adjacent kimberlite domains) to be used during estimation.

Ordinary Kriging (OK) was used to interpolate block estimates for the South Lobe domains, based on a single variogram model interpreted for the South Lobe. Inverse Distance Weighting (ID2) was used to interpolate block estimates of dry density for the Centre and North Lobes. Variogram and estimation parameters are summarized in Table 14-3 and Table 14-4, respectively.

Block estimation was conducted using two passes and search distances equal to the variogram range for the first pass, and 2 x the variogram range for the second pass. Search distances used for ID2 interpolation within the North and Centre Lobes were kept consistent with the variogram parameters interpreted for the South Lobe density data.

Table 14-3: South Lobe Dry Density Variogram Parameters

Lobe	Direction (degrees)			Nugget	Structure	Model	Sill	Range (m)		
	Dip	Dip Azimuth	Pitch					Major	Semi-Major	Minor
South	79	270	100	0.3	Structure 1	Spherical	0.28	105	70	85
					Structure 2	Spherical	0.42	225	140	100

Source: SRK (2023)

Table 14-4: Dry Density Estimation Parameters

Lobe	Method	Direction (degrees)			Estimation Pass	Min Samples	Max Samples	Max Samples Per Drillhole	Search Distance (m)		
		Dip	Dip Azimuth	Pitch					Major	Semi-Major	Minor
South	OK	79	270	100	Pass 1	6	12	4	225	140	100
					Pass 2	1	12	4	450	280	200
Centre and North	ID2	79	270	100	Pass 1	6	12	4	225	140	100
					Pass 2	1	12	4	450	280	200

Source: SRK (2023)

14.3 Grade Estimation

Diamond grade estimation has been conducted using two distinct methodologies:

- Local estimation of block grades based on large diameter drillhole (LDDH) bulk sample data; and
- Global estimation of diamond grade based on the correlation of microdiamond abundance with macrodiamond grade obtained from LDDH bulk sampling.

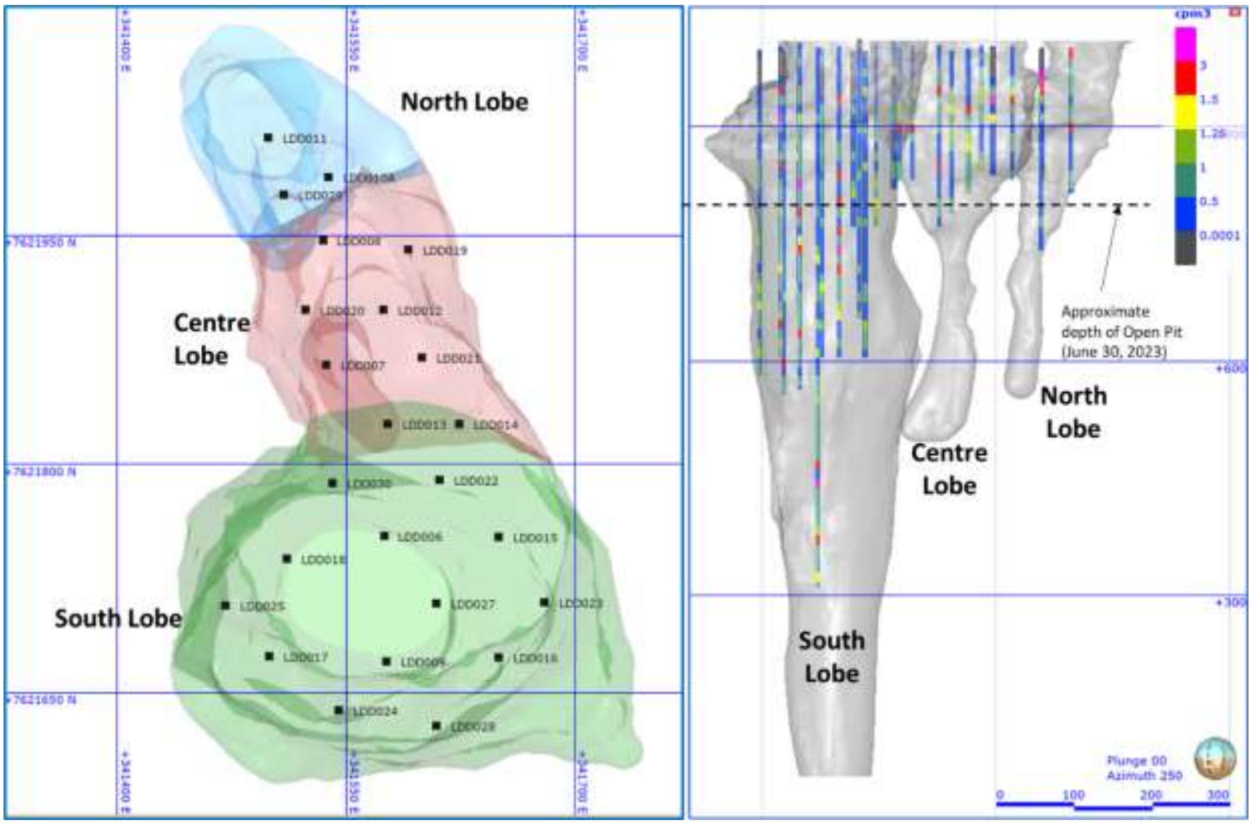
Global diamond grade estimation has solely been used within the deeper extents of South Lobe due to limited bulk sampling data available within this portion of the deposit.

14.3.1 Macrodiamond Data Summary

LDDH bulk sampling was conducted by De Beers in 2006 and 2007, during which time a 23-inch diameter rotary drill bit was used to complete 25 holes totalling 7,947 m of drilling. Holes were drilled vertically, and bulk samples were collected on nominal 12 m increments. All holes were caliper surveyed upon completion of drilling to determine sample volumes for each nominal 12 m sample interval.

Samples from 24 of the LDDH holes were processed at the time of the sampling campaigns and provide the macrodiamond data available for local grade estimation within the three lobes (Figure 14-5).

Figure 14-5: LDDH Bulk Sample Location Map and Sample Details



Note:
Sample grades color-coded by diamond grade expressed in carats per m^3 (cpm^3).

Source: SRK (2023)

A summary of the LDDH macrodiamond data is provided in Table 14-5, segregated according to the 2019 updated geological model. Note that the macrodiamond data has been segregated by internal domain for South Lobe only. No bulk sampling within the South Lobe KIMB3 domain has occurred to date.

The 2006 / 2007 bulk samples were initially processed at a De Beers bulk sample plant located outside of Letlhakane using a 10 t/hr DMS plant and concentrates were sent to the De Beers Group Exploration Macrodiamond Laboratory (GEMDL) in Johannesburg, South Africa, for final diamond recovery. All samples were processed using a +1.00 mm bottom cut-off.

Table 14-5: LDDH Bulk Sample Macrodiamond Data by Kimberlite Domain (+1.00 mm bottom cut-off)

DTC Sieve Class	EM/PK(S)		M/PK(S)		Centre		North	
	Carats	Stones	Carats	Stones	Carats	Stones	Carats	Stones
+23	0	0	7.98	2	13.37	1	0	0
+21	13.94	3	8.53	2	4.55	1	0	0
+19	14.62	6	30.27	14	15.17	7	2.27	1
+17	8.85	6	9.94	7	15.07	10	9.13	7
+15	6.96	7	3.62	3	9	8	2.35	3
+13	15.23	18	38.18	45	28.62	35	12.21	16
+12	13.36	24	22.89	44	11.29	21	10.01	17
+11	21.69	59	41.07	116	26.58	74	16.83	45
+9	33.98	165	60.69	295	38.51	187	15.54	76
+7	38.74	316	42.48	351	27.2	221	12.2	101
+6	33.13	368	38.64	445	22.26	250	11.33	128
+5	40.01	553	47.56	654	23.81	328	10.02	140
+3	51.65	1,478	53.4	1,532	31.49	902	8.72	253
+2	17.68	836	19.04	877	12.75	595	2.07	91
+1	10.76	769	13.56	967	7.59	545	1.74	129
Totals	**320.6**	**4,608**	**437.85**	**5,354**	**287.26**	**3,185**	**114.42**	**1,007**
Sample Volume (m^3)	**321.82**		**895.65**		**409.09**		**151.70**	
Sample Weight (t)	**887.7**		**2509.8**		**1018.7**		**374.8**	
Grade (cpht)	**36.1**		**17.4**		**28.2**		**30.5**	
Grade (cpm^3)	**1.00**		**0.49**		**0.70**		**0.75**	

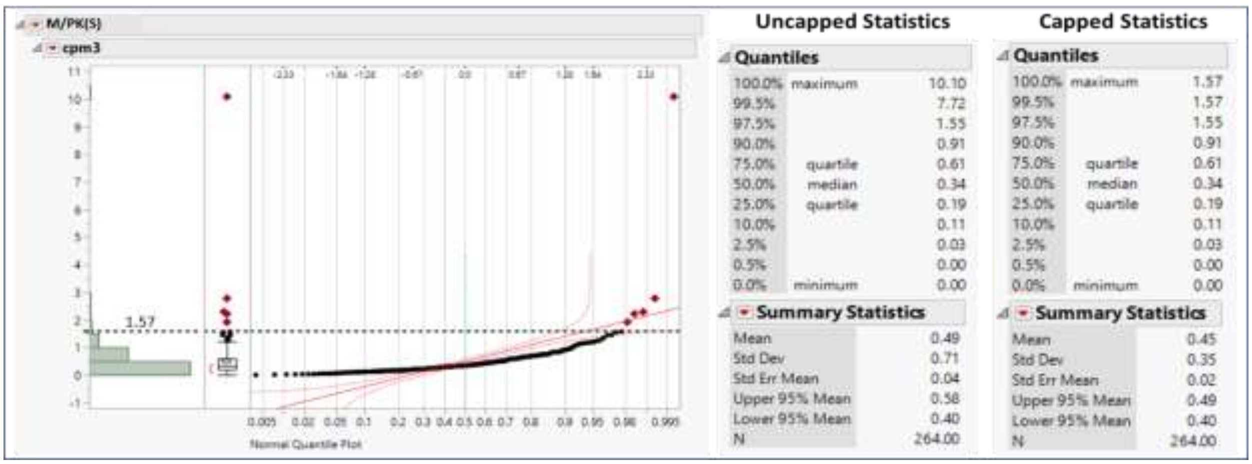
Source: SRK (2023)

14.3.2 Diamond Grade Capping Analysis

Based on historical production reconciliation for KDM, a grade capping analysis was conducted on the 2006 / 2007 LDDH bulk sample dataset for the South Lobe. Capping of anomalous high-grade samples (or outliers) is often required in "nuggety" deposits to minimize the influence these few samples can have during block grade interpolation.

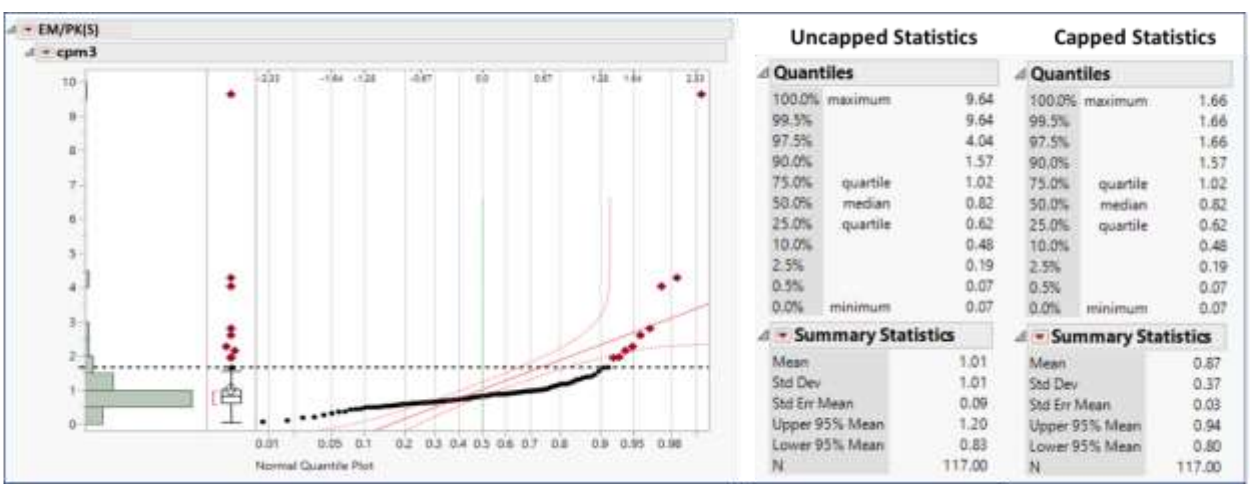
Figure 14-6 and Figure 14-7 provide details for the grade capping analysis for the South Lobe M/PK(S) and EM/PK(S) domains. Sample grades (expressed in units of cpm^3 (carats per cubic metre)) were plotted using a normal quantile plot and assessed for outliers, which have been highlighted as red diamonds on the figures below. For both the M/PK(S) and EM/PK(S) domains, anomalous high-grade samples were identified and capping values of 1.57 and 1.66 cpm^3 were selected, respectively. Sample summary statistics for uncapped and capped data populations are provided in the figures below. The capped datasets were used for subsequent diamond grade estimation.

Figure 14-6: South Lobe M/PK(S) Domain Grade Capping Analysis



Source: SRK (2023)

Figure 14-7: South Lobe EM/PK(S) Domain Grade Capping Analysis



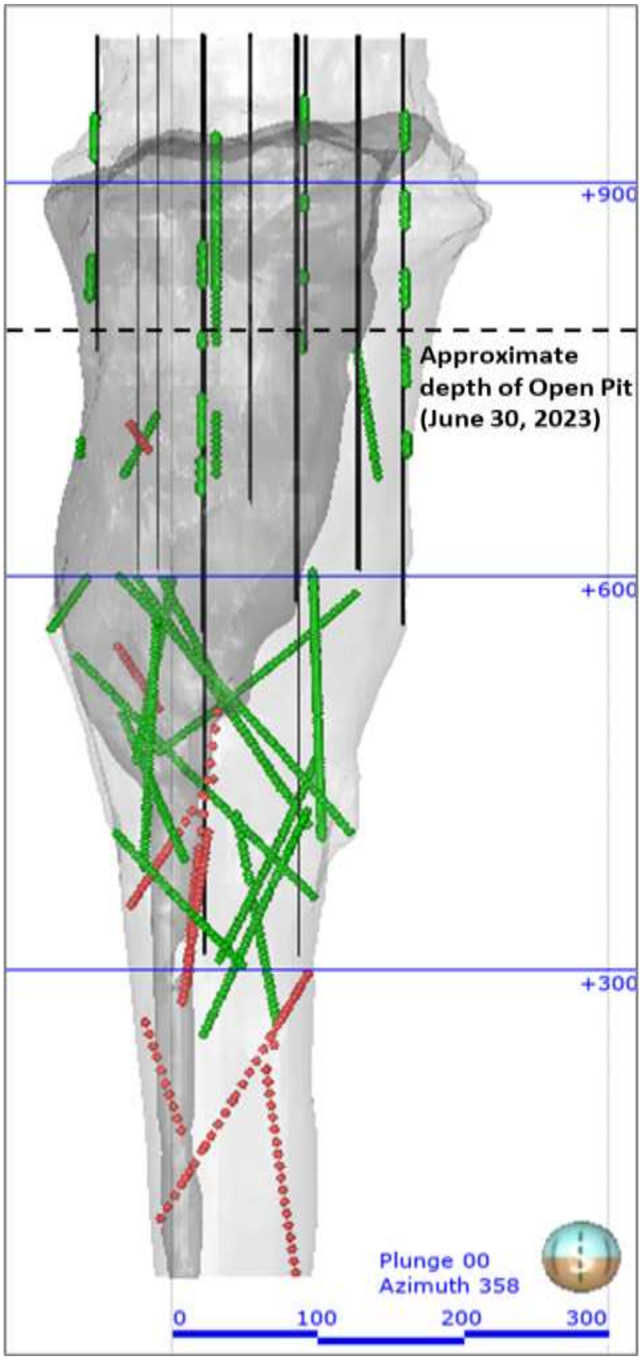
Source: SRK (2023)

14.3.3 Microdiamond Data Summary

The most recent microdiamond sampling within the South Lobe has been conducted in two sampling campaigns completed in 2017 and 2019, to assess diamond grade continuity within the deeper extents of the South Lobe below the LDDH bulk sample drilling (Figure 14-8). Historical microdiamond sampling (77 aliquots weighing 1,436 kg) was conducted prior to 2010, however due to data quality and reliability concerns this data has not been used within the current analysis. The 2017 sampling campaign was focused on representative sampling (from pilot core holes) of material drilled during the 2006 / 2007 LDDH campaign and deeper sampling of the two volumetrically dominant kimberlite domains within South Lobe (i.e., M/PK(S) and EM/PK(S)) between elevations 950 to 300 masl (Nowicki et al., 2018). The 2019 sampling campaign was focused on sampling of the volumetrically dominant EM/PK(S) domain between 450 to 70 masl, as well as sampling of the KIMB3 domain identified in 2019. A summary of the 2017 and 2019 microdiamond data is provided in Table 14-6, segregated by sampling campaign and kimberlite domain.

Microdiamond samples have been collected using nominal 8 kg aliquots of drill core and processed at the Saskatchewan Research Council (SRC) in Saskatoon, Saskatchewan, Canada. All samples have been processed using a bottom cut-off of +105 µm with total microdiamond recoveries per sieve class grouped by kimberlite domain summarized in Table 14-6.

Figure 14-8: Distribution of Microdiamond Samples



Note:
Sample collected from the South Lobe in 2017 (green) and in 2019 (red). Vertical black traces depict 2006 / 2007 LDDH bulk sample holes. M/PK(S) domain shown in dark grey, EM/PK(S) as lighter grey.

Source: SRK (2023)

Table 14-6: South Lobe Microdiamond Stone (Stns) Count Summary

	EM/PK(S)_2017	EM/PK(S)_2019	M/PK(S)_2017	KIMB3_2019
Sample Count	**464**	**98**	**374**	**39**
Dry Mass (kg)	**3,681.15**	**791.85**	**3,009.55**	**313.35**
Stns_+105	866	197	494	64
Stns_+150	603	110	258	39
Stns_+212	370	88	207	17
Stns_+300	271	59	127	19
Stns_+425	153	30	67	8
Stns_+600	102	24	34	1
Stns_+850	39	10	18	2
Stns_+1180	22	6	11	0
Stns_+1700	5	1	2	0
Stns_+2360	1	0	0	0
Stns_+3350	0	1	0	0
Total Stns	**2,432**	**526**	**1,218**	**150**
Stns/kg	**0.66**	**0.66**	**0.40**	**0.48**
Total Stns +150	**1,566**	**329**	**724**	**86**
Stns/kg +150	**0.43**	**0.42**	**0.24**	**0.27**

Source: SRK (2023)

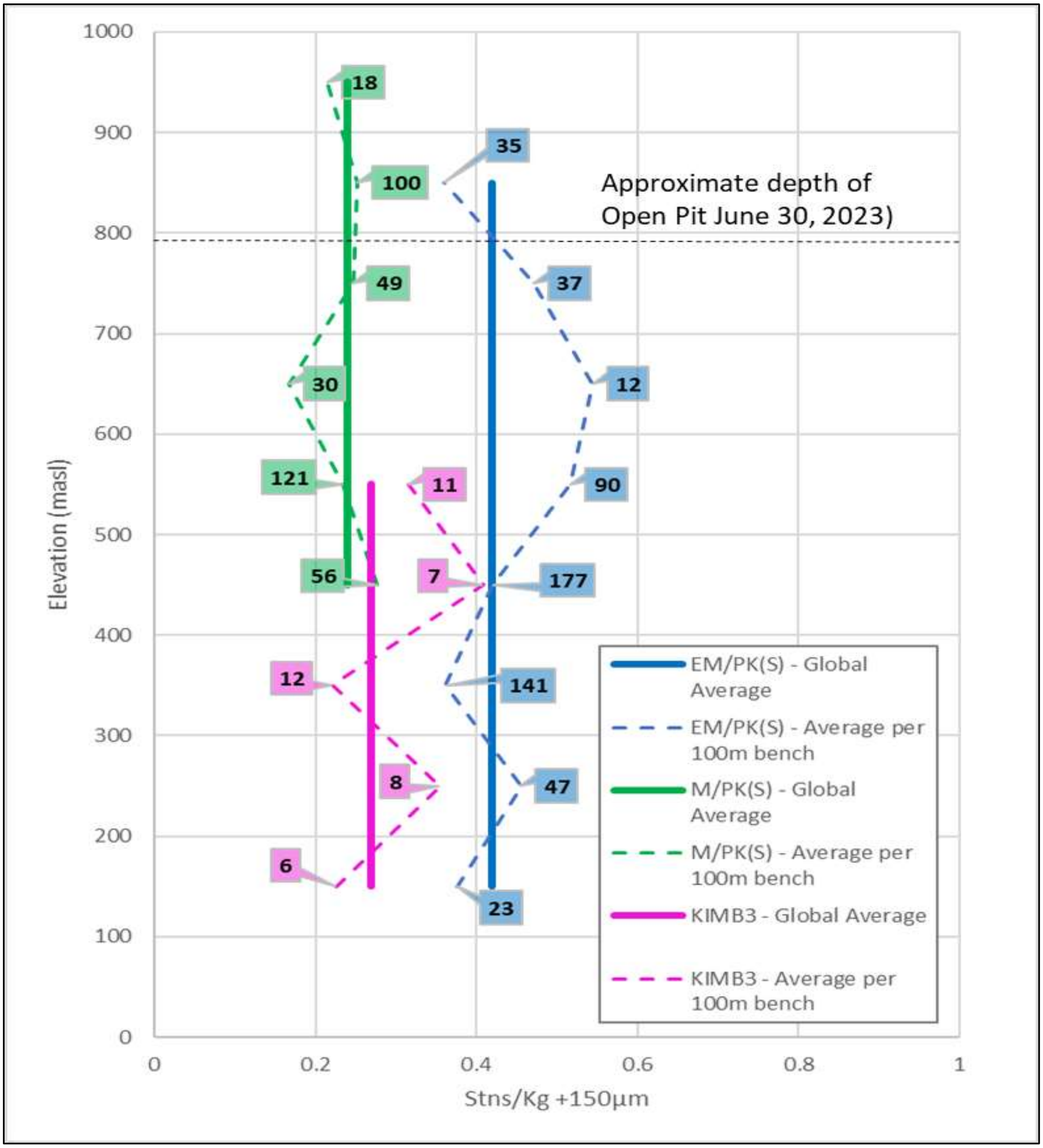
Similar microdiamond population statistics are observed between the 2017 and 2019 microdiamond datasets for the EM/PK(S) domain, as both sample groups have similar microdiamond stone densities (expressed as stones per kilogram, or Stns/kg) of 0.43 and 0.42 Stns/kg (larger than +150 µm), respectively. Figure 14-9 provides a comparison of the variable microdiamond stone density per 100 m vertical bench for the South Lobe internal domains, relative to each global average stone density. Notwithstanding the relatively small number of samples within some of the benches, broad continuity in stone density with depth is observed within both the EM/PK(S) and M/PK(S).

An SFD comparison for the EM/PK(S) 2017 and 2019 microdiamond populations is provided in Figure 14-10, which also demonstrates similar microdiamond population characteristics between the two sample groups. Therefore, no appreciable change in the microdiamond population within the EM/PK(S) domain occurs at depth and as such no significant change in the macrodiamond population characteristics is anticipated to occur at depth within the EM/PK(S) domain.

Comparison of microdiamond statistics between the EM/PK(S) and M/PK(S) domains demonstrates a material difference in mean stone density (i.e., 0.42 and 0.24 Stns/kg +150 µm, respectively) between these domains (Figure 14-9) and is reflective of the difference in macrodiamond grade between these domains (0.87 vs 0.45 cpm^3 recovered from LDDH bulk sampling) as provided in Sections 14.3.1 and 14.3.2. Figure 14-11 illustrates similar microdiamond SFDs for the South EM/PK(S) and M/PK(S) domains, notwithstanding the noted differences in microdiamond and macrodiamond content.

The limited microdiamond data obtained in 2019 for the KIMB3 domain provides a similar stone density to the M/PK(S) domain (Figure 14-9), however a finer SFD compared to both the South EM/PK(S) and M/PK(S) domains as depicted in Figure 14-11. As noted in Section 14.3.1, no bulk sampling of the KIMB3 domain has occurred to date and therefore no macrodiamond population is available for comparison with the microdiamond population.

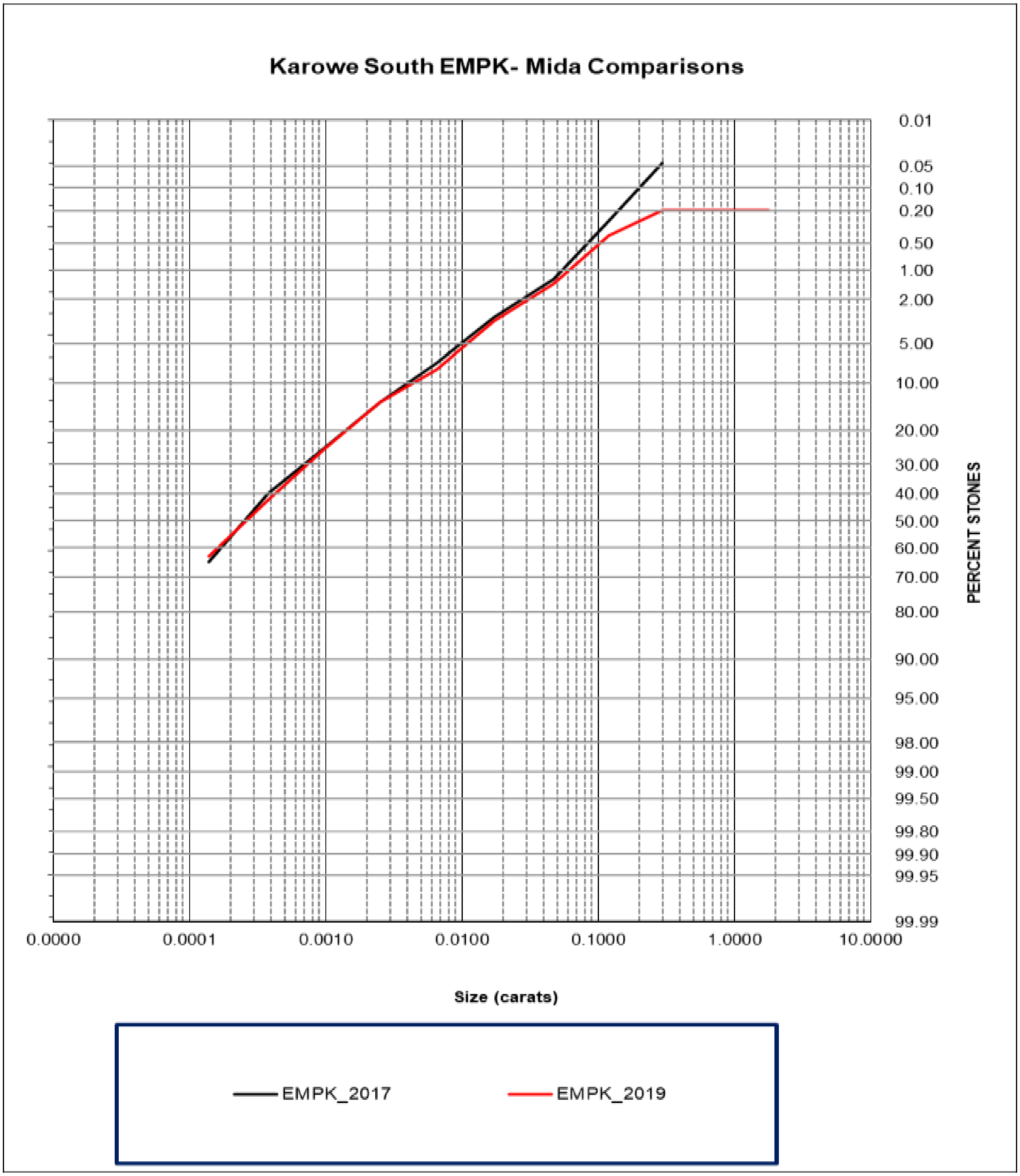
Figure 14-9: Comparison of Variable Microdiamond Stone Density per Kilogram



Note:
(+150 µm) per 100 m vertical benches for South Lobe internal kimberlite domains. Global domain averages are provided as solid lines. Values in callout boxes represent the number of 8 kg samples within each 100 m bench.

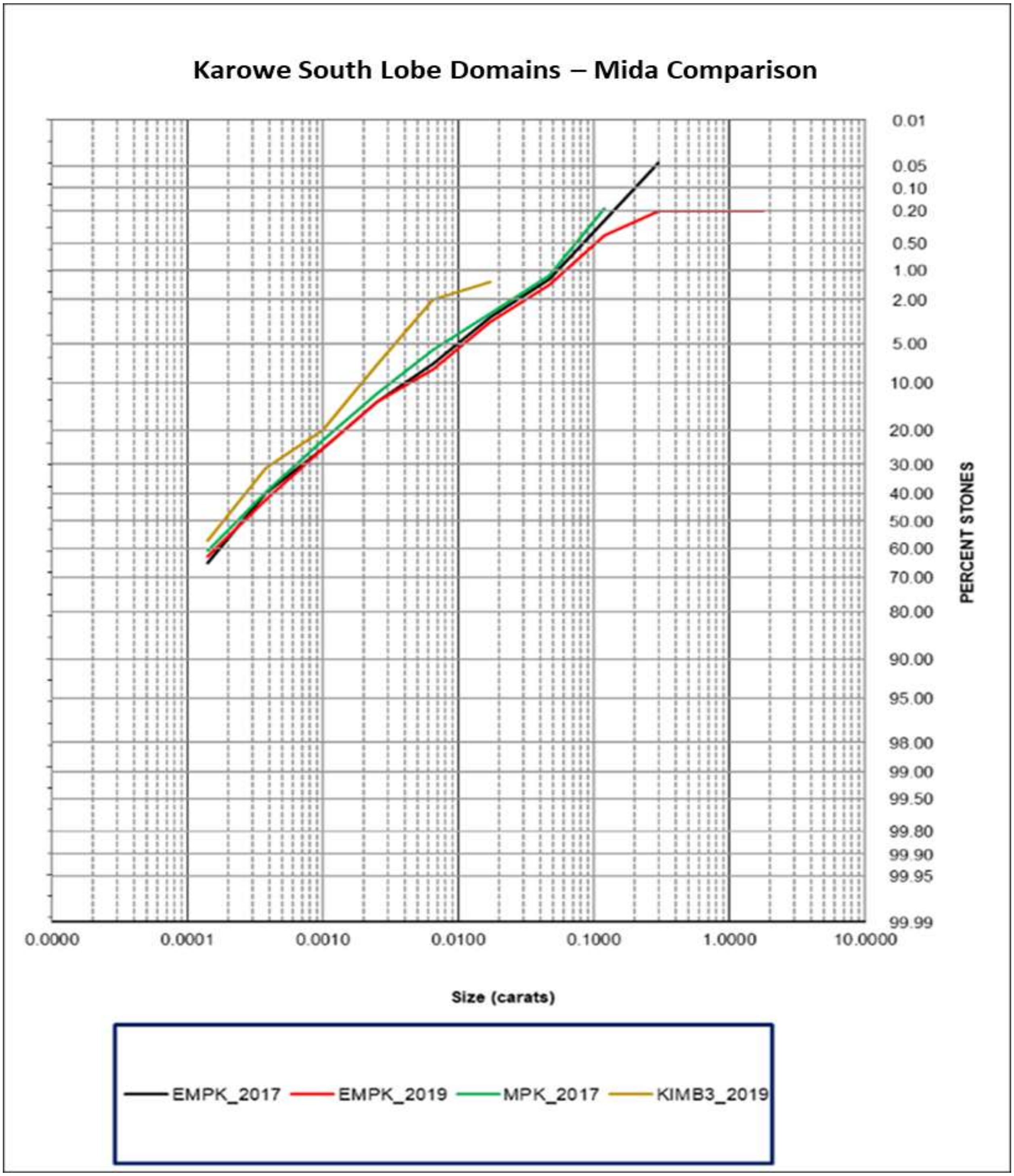
Source: SRK (2023)

Figure 14-10: South Lobe EM/PK(S) Microdiamond SFD Comparison



Source: SRK (2023)

Figure 14-11: South Lobe Internal Domain Microdiamond Populations SFD Comparison



Source: SRK (2023)

14.3.4 Local Grade Estimation

Similar to previous Mineral Resource Estimates completed in 2009, 2014, 2017 and 2018, a local grade estimation approach has been utilized where spatially representative LDDH bulk sample data is available. However, the approach employed in the current estimate has been modified to incorporate a hard boundary between the South Lobe M/PK(S) and EM/PK(S) domains due to the significant grade difference between these two domains. All previous Mineral Resource Estimates disregarded the contact between the M/PK(S) and EM/PK(S) domains, and therefore a single diamond grade dataset was used for local block estimation within the South Lobe. The current Mineral Resource Estimate is comprised of local diamond grade estimates to the depth of LDDH bulk sampling within the South Lobe M/PK(S) and EM/PK(S) domains at 604 and 568 masl, respectively.

As can be seen in Table 14-5, and Figure 14-6 and Figure 14-7, the average macrodiamond grade of the EM/PK(S) domain is approximately double the average macrodiamond grade of the M/PK(S) domain (36.1 vs 17.4 cpht recovered). The grade difference is consistent with diamond recoveries from discrete production samples of EM/PK(S) material mined from the OP within the last two years. Therefore, to produce a more robust local block grade estimate to support mine planning and production reconciliation, only diamond grade information located within each kimberlite domain was used to estimate block grades within that domain.

Block estimation for the South Lobe M/PK(S) and EM/PK(S) domains was conducted using OK. A single variogram model for diamond grade (expressed as cpm^3) was developed for the South Lobe due to the limited number of samples available from the LDDH bulk sampling campaigns (Table 14-7).

Table 14-7: South Lobe Diamond Grade Variogram Model

Direction (degrees)			Nugget	Structure	Model	Sill	Alpha	Range (m)		
Dip	Dip Azimuth	Pitch						Major	Semi-Major	Minor
0	0	65	0.07	Structure 1	Spheroidal	0.245	3	110	90	40

Source: SRK (2023)

North and Centre Lobe diamond grade estimation was conducted using ID2, using a hard boundary for both lobes to isolate their respective diamond grade populations. Parameters used for local diamond grade estimation are provided in Table 14-8. A two-pass approach was followed, such that blocks not estimated using Pass 1 parameters were estimated using the Pass 2 parameters. Sample search distances of 1.0 x and 1.4 x the variogram range (along the horizontal axis) were used for Pass 1 and Pass 2, respectively. Centre and North Lobe estimation parameters were kept consistent with South Lobe parameters. The vast majority of blocks were estimated during Pass 1, with only a small proportion of blocks located along the margins of the kimberlite domains estimated during Pass 2.

Table 14-8: Diamond Grade Estimation Parameters

Lobe	Method	Search Direction (degrees)			Estimation Pass	Min Samples	Max Samples	Max Samples Per Drillhole	Search Distance (m)		
		Dip	Dip Azimuth	Pitch					Major	Semi-Major	Minor
South	OK	0	0	65	Pass 1	4	12	3	110	90	40
					Pass 2	1	12	3	150	125	80
Centre and North	ID2	0	0	65	Pass 1	4	12	3	110	90	40
					Pass 2	1	12	3	150	125	80

Source: SRK (2023)

14.3.5 Global Grade Estimation

A global grade estimation approach within the deeper portion of South Lobe (below 604 and 568 masl for M/PK(S) and EM/PK(S) domains, respectively) has been incorporated into the 2019 Mineral Resource update. The methodology is based on establishing a relationship between microdiamond stone abundance and macrodiamond grade within each kimberlite domain and demonstrating consistency in the geology and microdiamond data populations with depth.

As previously summarized in Sections 14.3.1 and 14.3.3, the relative difference in macrodiamond grade between the EM/PK(S) and M/PK(S) domains of 0.87 cpm^{3}and 0.45 cpm^3 (+1.0 mm bottom cut-off) respectively, is mirrored in microdiamond stone densities of 0.43 and 0.24 Stns/kg +150 µm, respectively, from the 2017 microdiamond sampling campaign. Furthermore, the 2019 microdiamond stone density within the EM/PK(S) domain (i.e., 0.42 Stns/kg +150 µm) at depth is consistent with the 2017 microdiamond population (Figure 14-9) and supports the projection of a consistent macrodiamond grade (+1.0 mm bottom cut-off) at depth.

The KIMB3 domain has been assigned a macrodiamond grade consistent with the M/PK(S) domain based on the following two assumptions:

- Microdiamonds from KIMB3 have a similar SFD as microdiamonds from the M/PK(S) domain (Figure 14-11). The ratio of micro- to macrodiamonds obtained for M/PK(S) material is hence assumed applicable to KIMB3; and

- A microdiamond stone density of 0.24 Stns/kg +150 µm for M/PK(S) correlates with a +1.0 mm macrodiamond content of 0.45 cpm^3.

As noted earlier, no bulk sampling of the KIMB3 domain has been conducted to date. There is a significant amount of uncertainty with the macrodiamond grade projection for the KIMB3 domain, and this has been considered in the Mineral Resource classification for this domain.

14.3.6 Adjustment for Production Plant Recovery Efficiency

The LDDH bulk sample data obtained in 2006 / 2007 and used for local grade estimation was processed using a nominal +1.0 mm bottom size cut-off. However, the configuration of the KDM processing plant uses a nominal +1.25 mm bottom cut-off for diamond recovery and therefore estimated grades based on the LDDH data requires adjustment to compensate for this larger bottom cut-off. The previous production plant recovery factor used to adjust +1.0 mm grades to +1.25 mm grades was -30%, determined from an SFD comparison of discrete production from South Lobe collected in March 2018 relative to the LDDH data.

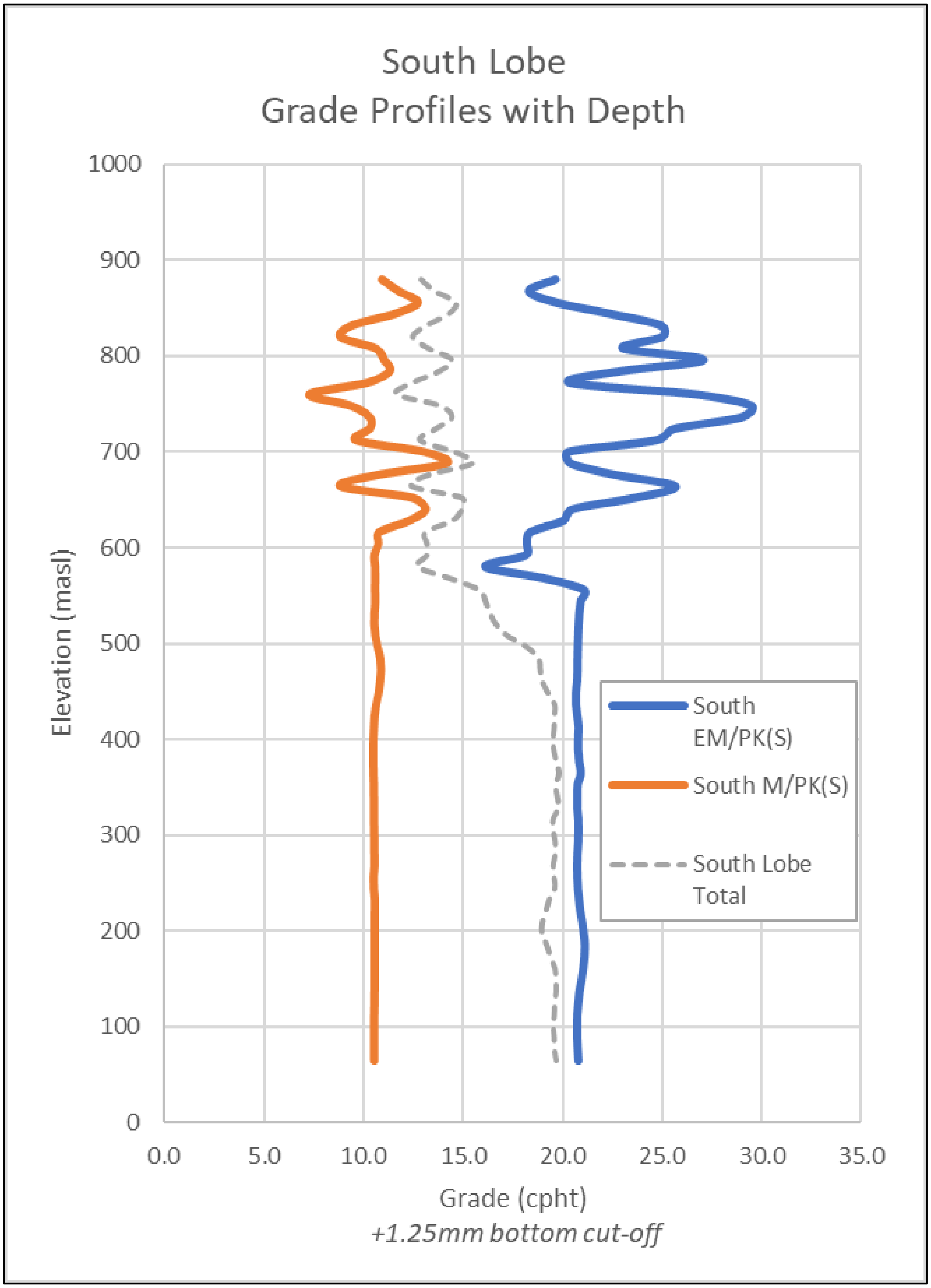
Over the course of 2018 and 2019, modifications within the KDM process plant improved the recovery efficiency of smaller diamonds within the mine production. Based on a comparison of quarterly mine production from Q4 2017 to Q3 2019, adjustment to the process recovery factor was required to reflect increased recovery of diamonds within the -7 DTC sieve size fractions. A process recovery factor of -28.5% has been used to adjust nominal +1.0 mm bottom cut-off grade estimates to +1.25 mm bottom cut-off grade estimates for the 2019 Mineral Resource update.

14.3.7 Grade Estimation Summary

Vertical profiles of recoverable grade (cpht) at a bottom cut-off of +1.25 mm for the South Lobe are provided in Figure 14-12. The profiles represent the grade estimation approach adopted for this Mineral Resource Estimate and reflect variable local grade estimates supported by LDDH bulk sample data shallower than approximately 570 masl. The near-constant grades estimated deeper than 570 masl reflect a global grade estimation approach, underpinned by the calibrated relationship of micro- to macrodiamond content and representative microdiamond sampling within the deeper portions of the Lobe. The “South Lobe Total” profile in Figure 14-12 reflects a combined grade profile for the entire South Lobe (including the KIMB3 domain), weighted by tonnages of each kimberlite domain per 12 m vertical bench intervals.

Figure 14-12 illustrates that total recoverable grade in the South Lobe increases from approximately 14 cpht at 580 masl to approximately 19 cpht at 450 masl and deeper, due largely to the higher-grade EM/PK(S) domain expanding to occupy around 87 percent by volume of the South Lobe over the interval 420 to 70 masl.

Figure 14-12: Recoverable Grade Profile with Depth for the Dominant South Lobe Domains



Source: SRK (2023)

14.4 Diamond Value Estimate

Diamond value estimates presented in this section have been generated by Lucara and are based on LOM production and sales information to the end of June 2023. The diamond value estimates incorporate current trends observed through diamond tenders, Clara and HB Antwerp sales data along with production data from KDM and are representative of the current status of the diamond market at the effective date. Mr. Revering has reviewed the information and analysis provided by Lucara and considers them to be reliable and consistent with average US$ per carat prices disclosed in Lucara quarterly financials.

Diamond value estimates are the product of the size frequency distribution of a given diamond population and the diamond quality characteristics of that population; and are typically unique for each kimberlite domain within a deposit. The 2023 Mineral Resource Estimate for KDM incorporates unique diamond value estimates for the two main kimberlite domains within the South Lobe (i.e., M/PK(S) and EM/PK(S)) based on discrete production and diamond sales data obtained from these domains and better reflects the reconciled production data. The North and Centre Lobe diamond value estimates have slight price improvements based on current market conditions versus the previous models from 2019. In addition, the SFD model for the M/PK(S) unit has been modified to better reflect reconciled production data, resulting in a slightly coarser model and therefore value increase.

14.4.1 Size Frequency Distribution Model

Details of the discrete production parcels used to develop SFD models for the North and Centre Lobes, and the South Lobe M/PK(S) and EM/PK(S) domains are provided in Table 14-9. Prior to 2019, a single diamond SFD model was used for the entirety of South Lobe because of limited discrete production data available for the EM/PK(S) domain due to its lack of exposure near surface. However, over the course of 2018 and 2019, mine production from the EM/PK(S) domain was possible allowing for the development of a distinct SFD model. Reconciliation of the volume of +10.8 ct diamonds recovered versus the weighted modelled volumes indicated that the SFD model for the M/PK(S) was too conservative, adopting a coarser SFD model that aligns with the actual +10.8 ct weight percent for the large (409 k ct) M/PK(S) sample results in better reconciliation to production data (Table 14-9). M/PK(S) carat contributions were the dominant source of carats and feed to plant over the period shown in Table 14-10 and therefore alignment to the large M/PK(S) sample generates good reconciliation between model and actual volumes of +10.8 diamonds. SFD models for the North and Centre lobes are unchanged from the 2019 FS and 2018 resource update. It should be noted that for the EM/PK(S) domain, the SFD model slightly underestimate the percentage of the +10.8 ct size class compared to the actual production parcels. This impact is discussed further in Section 14.4.2.

A comparison between the 2019 South Lobe SFD model and 2023 SFD models for the M/PK(S) and EM/PK(S) domains is provided in Table 14-11. The most significant change to note in these SFD models is within the +10.8 ct size fraction, which is associated with the most significant revenue component of KDM production as further discussed in Section 14.4.2.

Table 14-9: Annual Diamond Mass and Size Distributions by Kimberlite Lobe/Type

Year	Proportion Carats Recovered					Weight Percent +10.8ct		
	EM/PK(S) (%)	M/PK(S) (%)	North (%)	Centre (%)	Sum (%)	2019 Model (%)	Actual (%)	2023 Model** (%)
2020	41	56	0	3	100	6.69	6.7	6.91
2021	35	64	0	0.4	100	6.64	7.8	6.89
2022	24	76	-	-	100	6.41	6.7	6.70
2023*	20	42	6	32	100	5.11	5.2	5.78

Notes:
*To end Q2/2023
**2023 Model uses actual recovered +10.8ct for M/PK(S) (6.3 wt.% vs FS SFD of 5.91 wt.%) refer to Table 14-10.

Source: Lucara (2023)

Table 14-10: Discrete Production Parcel Data for North Lobe, Centre Lobe, and South Lobe

Size Class	Discrete Production Parcels (cts per size class)				Discrete Production Parcel SFD's (% cts per size class)				2023 Model SFDs (% cts per size class)			
	M/PK(S)	EM/PK(S)	Centre	North	M/PK(S)	EM/PK(S)	Centre	North	M/PK(S)	EM/PK(S)	Centre	North
+10.8 ct	25,802	3,933	8,836	579	6.3	8.3	3.4	1.0	6.3	8.0	3.1	1.0
6-10 ct	11,852	1,417	5,626	1,140	2.9	3.0	2.2	2.0	2.89	3.6	2.9	2.4
3-5 ct	23,854	2,739	14,378	3,552	5.8	5.8	5.6	6.2	5.82	5.6	3.9	5.3
8-10 gr	22,166	2,156	14,263	4,058	5.4	4.6	5.5	7.1	5.41	4.1	7.2	7.7
3-6 gr	71,559	6,410	50,292	14,732	17.5	13.6	19.6	25.7	17.47	14.0	19.4	25.7
+11DTC	75,466	7,695	53,852	14,130	18.4	16.3	20.9	24.7	18.4	16.3	21.0	24.7
+9DTC	62,232	6,763	41,516	9,116	15.2	14.4	16.1	15.9	15.2	14.4	15.9	15.9
+7DTC	46,027	5,150	28,524	5,288	11.2	10.9	11.1	9.2	11.2	10.9	11.0	9.2
+5DTC	62,701	8,892	36,214	4,584	15.3	18.9	14.1	8.0	15.3	18.9	14.0	8.0
+3DTC	7,985	1,949	3,686	73	1.9	4.1	1.4	0.1	2.0	4.1	1.3	0.1
Total Carats	**409,644**	**47,103**	**257,187**	**57,252**								

Note:
Size class abbreviations are “DTC” = Diamond Trading Company, “gr” = grainer, and “ct” = carats and resultant SFD models at +1.25mm bottom cut-off.

Source: Lucara (2023)

Table 14-11: Comparison of 2019 and 2023 SFD Models for South Lobe

Size Class	SFD Models (% cts per size class)		
	M/PK(S) 2019	M/PK(S) 2023	EM/PK(S) 2019/23
+10.8 ct	5.9	6.3	8.0
6-10 ct	3.5	2.9	3.6
3-5 ct	5.8	5.8	5.6
8-10 gr	4.5	5.4	4.1
3-6 gr	18.2	17.5	14.0
+11 DTC	18.4	18.4	16.3
+9 DTC	15.2	15.2	14.4
+7 DTC	11.2	11.2	10.9
+5 DTC	15.3	15.3	18.9
+3 DTC	2.0	2.0	4.1

Source: Lucara (2023)

14.4.2 Value Distribution Models

The 2023 value distribution models are provided in Table 14-12, and are based on discrete mine production data for each kimberlite domain obtained since the start of mining and diamond sales information to the end of March 2023. The average US$/ct estimates for each of the main ore sources (North, Centre, M/PK(S) and EM/PK(S) have been revised based on sales data since 2019. The sales mechanism for KDM diamonds prior to 2019 was through closed Tenders, since 2019 the sales mechanism has three distinct channels, closed tender, Clara platform and HB offtake agreement (see Section 19.1 for details).

All diamonds less than 10.8 ct in weight are sold either via tenders, or those stones in the range from 6 gr to 10.8 ct in the better qualities via the Clara platform. The 2019 FS average price assumption for -10.8 ct diamonds was $190/ct. Sales data for 2021 and 2022(excluding first sales of 2021) indicate that the -10.8 ct average price (Tender + Clara) is approximately 19% greater than the 2019 assumptions. Price models (-10.8 ct) for each ore source have been adjusted upward by 19% over the 2019 values for the 2023 value model. Diamonds greater than 10.8 ct are sold via Tender (rejection/board/low cleavage) and through the HB offtake agreement. Based on sales results the price point for +10.8 ct for M/PK(S) and EM/PK(S) has increased by approximately 1.4% from the 2019 FS model ($7600/ct) to $7706/ct for the new 2023 model. The Centre Lobe price point has decreased from $6225/ct to $5600/ct. In addition, the SFD model for the M/PK(S) has coarsened resulting in an additional average price per carat increase for this unit. As shown in Table 14-12, the average value per size class for the M/PK(S) and EM/PK(S) domains are very similar and reflects similar diamond quality characteristics between these two domains. However, the overall higher average US$/ct for the EM/PK(S) domain reflects the coarser diamond SFD for this domain specifically within the +10.8 ct size fraction.

Table 14-12 details the changes to the price per carat for each of the main ore sources.

Table 14-12: 2019 vs. 2023 Feasibility Study Diamond Price Assumptions

Source	2019 FS AP ($)	Adjustment			2023 AP/ct ($)	Variance (%)
North	221	-10.8ct $/ct increase based on 2021/22 tenders and Clara sales			273	24
Centre	349	-10.8ct $/ct increase based on 2021/22 tenders and Clara sales	decrease price point for Centre Lobe +10.8ct		392	12
EM/PK(S)	777	-10.8ct $/ct increase based on 2021/22 tenders and Clara sales	increase price point for +10.8ct by 1.4%		828	7
M/PK(S)	631	-10.8ct $/ct increase based on 2021/22 tenders and Clara sales	increase price point for +10.8ct by 1.4%	coarser SFD	707	12

Source: Lucara (2023)

Table 14-13: 2023 Value Distribution Models for KDM

Size Class	2023 Model SFD's (% cts per size class)				2023 % Revenue per size class			EM/PK(S) % Rev
	North (%)	Centre (%)	M/PK(S) (%)	EM/PK(S) (%)	North % Rev	Centre % Rev	M/PK(S) % Rev	
+10.8 ct	1.0	3.1	6.31	8.0	6	45	69	74
6-10 ct	2.4	2.9	2.89	3.6	12	11	6	6
3-5 ct	5.3	3.9	5.82	5.6	19	8	7	6
8-10 gr	7.7	7.2	5.4	4.1	17	10	4	3
3-6 gr	25.7	19.4	17.5	14.0	26	13	7	5
+11 DTC	24.7	21.0	18.4	16.3	11	6	3	2
+9 DTC	15.9	15.9	15.2	14.4	5	3	2	2
+7 DTC	9.2	11.0	11.2	10.9	2	2	1	1
+5 DTC	8.0	14.0	15.3	18.9	2	2	1	1
+3 DTC	0.1	1.3	2.0	4.1	0.02	0.2	0.1	0.2
					100	100	100	100
				$/ct	$ 273	$ 392	$ 707	$ 828

Source: Lucara (2023)

As mentioned in Section 14.4.1, the modelled SFD's for the South Lobe EM/PK(S) domain slightly underestimate the proportion of +10.8 ct diamonds when compared to the actual production diamond SFD's as shown in Table 14-11. The impact on the *average* US$/ct for the EM/PK(S) domains is a reduction of $20/ct compared against actual production SFD. Diamond prices used in the 2023 Mineral Resource Estimate accordingly reflect a conservative value model compared to actual production. No diamond price escalation is incorporated into the price assumptions.

Value models exclude from the pricing approximately $260 M in revenue generated from +$10 M single stones (i.e., exceptional stones) sold since 2014, which includes such diamonds as the Constellation (813 ct sold for $63 M) and the Lesedi la Rona (1,109 ct sold for $53 M). Revenues from the sale of such exceptional diamonds vary materially through time, though represent approximately 11 percent of all diamond sales revenue since the start of commercial production in April 2012. Total sales of approximately 3.99 Mct since the start of commercial production have generated revenue of $2.23 B, for a LOM average price per carat of $558/ct (including exceptional stones). The South Lobe consistently recovers high value diamonds in excess of 200 ct in size with a periodicity of, on average every 5 quarters, large high value potential exceptional diamonds are recovered. For example, the 549 ct recovery of 2020 (unsold), the 1174 ct diamond of 2021, and the 1080 ct diamond recovered in Q3/2023.

The KIMB3 domain has been assigned an average $/ct value consistent with the M/PK(S) domain, based primarily on a similar microdiamond SFD (Section 14.3.3). There is currently no macrodiamond parcel available from the KIMB3 domain by which to assess quality and value characteristics. Therefore, a significant amount of uncertainty is associated with the value projection for the KIMB3 domain, which has been considered in the Mineral Resource classification for this domain.

14.5 Mineral Resource Statement and Classification

A Mineral Resource is defined by the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) as:

> *"a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."*

CIM further defines "reasonable prospect of eventual economic extraction" as:

> *"a judgment in respect of the technical and economic factors likely to influence the prospect of economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs."*

The 2023 Mineral Resources for the KDM have been classified as either Indicated or Inferred Mineral Resources. No Measured Mineral Resource has been defined for this deposit. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) define Indicated and Inferred Mineral Resources as follows:

Indicated Mineral Resource

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

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

The two dominant kimberlite domains within the South Lobe (i.e., M/PK(S) and EM/PK(S)) have been classified as Indicated Mineral Resources to a depth of 250 masl, based on drillhole coverage, geological continuity and available sample information (i.e., petrography-control, bulk density, microdiamond and macrodiamond data) as documented in previous sections of this report. Below 250 masl, both the M/PK(S) and EM/PK(S) domains are classified as Inferred Mineral Resource. The KIMB3 domain is entirely classified as Inferred Mineral Resources due to insufficient diamond data to support an assessment of macrodiamond grade and value characteristics within this kimberlite domain, and limited drillhole coverage to adequately assess geological continuity at higher confidence levels. Both the North and Centre Lobes are classified as Indicated Mineral Resources to depths of 745 masl.

The 2023 Mineral Resource statement for KDM (with an effective date of June 30, 2023) is provided in Table 14-14, which is inclusive of Mineral Reserves.

Table 14-14: KDM 2019 Mineral Resource Statement

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mcts)	Grade (cpht)	Average $/ct
Indicated	South_M/PK(S)	7.02	20.92	2.96	2.27	10.8	$707
	South_EM/PK(S)	6.77	19.77	2.90	4.16	21.0	$828
	Centre	0.30	0.81	2.57	0.12	15.5	$392
	North	0.18	0.42	2.45	0.05	11.6	$273
Total Indicated		**14.27**	**41.92**	**2.90**	**6.60**	**15.8**	**$793**

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mcts)	Grade (cpht)	Average $/ct
Inferred	South_M/PK(S)	0.10	0.31	3.05	0.03	10.5	$707
	South_EM/PK(S)	1.40	4.18	2.97	0.87	20.9	$828
	South_KIMB3	0.32	0.94	2.94	0.10	10.9	$707
Total Inferred		**1.82**	**5.42**	**2.97**	**1.01**	**18.6**	**$804**

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All numbers have been rounded to reflect accuracy of the estimate.
2. Mineral Resources are in-situ Mineral Resources and are inclusive of in-situ Mineral Reserves.
3. Mineral Resources are exclusive of all mine stockpile material.
4. Mineral Resources are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond losses within the smaller sieve classes expected within a commercial process plant.
5. Inferred Mineral Resources are estimated on the basis of limited geological evidence and sampling, sufficient to imply but not verify geological grade and continuity. They have a lower level of confidence than that applied to an Indicated Mineral Resource and cannot be directly converted into a Mineral Reserve.
6. Average diamond value estimates are based on 2023 diamond sales data provided by Lucara Diamond Corp.
7. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery.

(Effective date of June 30, 2023)

Source: SRK (2023)

Mr. Revering is not aware of any environmental, permitting, legal, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the Mineral Resource Estimate other than those discussed in the report.

15 MINERAL RESERVE ESTIMATE

15.1 Mining Method

A consolidated OP and UG mine plan were developed to extract the economic portions of the KDM Indicated Mineral Resources plus stockpiled ore. The mine plan includes extraction of three adjacent lobes of kimberlite. The South Lobe is planned to be mined through a combination of OP and UG mining methods. The Centre Lobe is planned for extraction by OP mining methods only. The North Lobe mined from the OP, is uneconomic and not considered a reserve.

15.2 Mining Dilution

15.2.1 OP Dilution

A total OP dilution of 0.0% has been included in the OP reserve estimate. This assumption is consistent with current operations and has been applied historically on the project.

15.2.2 UG Dilution

A total UG dilution of 9.6%, or 3.5 Mt has been included in the UG reserve estimate. Two types of UG dilution were applied to the stope and development designs:

- External Dilution; and
- Internal Dilution.

15.2.2.1 External Dilution

External dilution accounts for additional material (overbreak) that is mined outside of the resource. This material is mined with zero grade and value assigned to it. External dilution estimates have been defined by geotechnical rock mass domains, stope strike length and dip, and mining method.

The large, continuous nature of the resource combined with excellent ground conditions in both the kimberlite and most of the host rock suggests little to no dilution will occur in the granite lithology domains. However, a 1.0 m dilution halo has been included in all stope designs to account for production drilling inaccuracies. Above the granite, geomechanical modelling has predicting approximately 2.7 Mt of host rock falling into the stope once exposed. The combined value of both these external dilutions estimates is 3.2 Mt or 8.7% of the UG reserve.

15.2.2.2 Internal Dilution

Internal dilution, or designed dilution, accounts for additional, lower than cut-off value (COV) material within the planned stope or development design shape. Grades for internal dilution are taken from the Mineral Resource model if available otherwise they are assigned a value of zero.

Any Inferred Resource class material within the mining reserve stope and development shapes has been treated as waste and has been assigned zero value. Inferred dilution comprises approximately 330 kt or 0.9% of the UG reserve.

15.3 Mining Recovery

A 100% mine recovery has been assumed for the OP and UG reserves. This assumption has been applied during the operations since the onset of the project.

15.4 Process Recovery

Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the Mineral Resource block model assumptions and therefore have taken recoveries into account.

15.5 Cut-off Value Criteria

The three adjacent lobes of kimberlite have varying diamond value per carat as outlined in Section 14.4. A cut-off value by mining method is used to calculate the mining reserve, instead of determining a specific cut-off grade for each kimberlite lobe.

Operating costs were estimated from existing operational charges, previous studies, and future forecasts. The cut-off values consist of estimated operating costs from three key areas:

- Mining – Costs vary by OP, UG, and stockpile operations;
- Processing – Processing costs are consistent for all materials; and
- G&A – Inclusive of cost of sales and corporate charges (Botswana). G&A costs are assumed to be reduced during stockpile processing after the completion of mining operations.

Rock value is calculated from diamond valuation, payable content, royalties, mining dilution, mining recovery, and process recovery parameters. The rock value must exceed the established cut-off value in the Mineral Reserve Estimate.

The COV parameters are shown in Table 15-1.

Table 15-1: Cut-Off Value Parameters

Parameter	Unit	Value		
		Open Pit	UG	Stockpile
Revenue, Smelting and Refining				
Payable content	%	100.0		
Royalty	%	10.0		
Mining Recovery and Dilution				
Mining Recovery	%	100.0		
Mining Dilution	%	0.0	9.6	N/A
Processing Recovery	%	100.0		
Operating Costs				
Mining	$/t milled	13.00	11.00	2.00
Processing	$/t milled	12.00		
G&A	$/t milled	12.00	12.00	5.00
Cut-Off Value	**$/t milled**	**37.00**	**35.00**	**19.00**

Source: JDS (2023)

15.6 Mineral Reserve Estimate

All Mineral Reserves are classified as Probable.

The Qualified Person preparing the Mineral Reserve Estimate, Brandon Chambers, P.Eng., did not identify any extraordinary risk, including legal, political, or environmental that would materially affect potential Mineral Reserves development. The effective date of this Mineral Reserve Estimate is Jun. 30, 2023.

Table 15-2: KDM Mineral Reserve Estimate (Jun. 30, 2023)

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	LOM Diamond Price
		(Mt)	('000s ct)	(cpht)	(US$/ct)
Open Pit					
Centre	Probable	0.6	96	16.3	392
South - EM/PK(s)	Probable	1.3	323	25.4	828
South - M/PK(s)	Probable	3.6	384	10.7	707
Open Pit	**Total**	**5.5**	**803**	**14.7**	**718**

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	LOM Diamond Price
		(Mt)	('000s ct)	(cpht)	(US$/ct)
UG					
South - EM/PK(s)	Probable	18.6	3,361	18.1	828
South - M/PK(s)	Probable	18.4	1,871	10.2	707
UG	**Total**	**37.0**	**5,232**	**14.2**	**785**
Stockpile					
Mixed Stockpile	Probable	4.0	502	12.7	433
Life of Mine	Probable	5.8	296	5.1	574
Stockpile	**Total**	**9.7**	**798**	**8.2**	**485**
Combined					
All	**Total**	**52.2**	**6,834**	**13.1**	**742**

Notes:

1. Prepared by Brandon Chambers, P.Eng. JDS Energy & Mining Inc.;
2. CIM definitions were followed for Mineral Reserves;
3. Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the Mineral Resource block model assumptions and therefore have taken recoveries into account;
4. The bottom elevation of the Probable Reserve is 310 masl;
5. Mineral Reserves are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond loses within the smaller sieve classes expected within the current configuration of the KDM Process Plan;
6. Diamond price estimates are provided by Lucara; prices are derived from historical sales and adjusted for current market conditions;
7. Tonnages are rounded to the nearest 100,000 t, diamond grades are rounded to one decimal place to properly reflect the Reserve estimate accuracy;
8. Tonnage and grade measurements are in metric units; contained diamonds are reported as thousands of carats;
9. Open Pit Mineral Reserves are estimated at a cut-off value of $37/t based on an OP mining cost of $13/t, a processing cost of $12/t and a G&A cost of $12/t;
10. UG Mineral Reserves are estimated at a cut-off value of $35/t based on an UG mining cost of $11/t, a processing cost of $12/t and a G&A cost of $12/t;
11. Mine Call Factor is a modifying factor used by Lucara which tracks the reconciliation between the block model and actual recovered carats. Mine Call Factor is assumed to be 100%, historically this factor has reconciled either near or above 100%, however in the 12-month period prior to the Reserve Statement the Mine Call Factor has deviated away from historical average performance and is currently at 95%;
12. UG dilution assumptions in the 2019 FS were revised in 2023. UG dilution included in the Reserve was estimated from the following three sources:
    - 1.0 m of zero-grade overbreak from stoping adjacent to the granite host rock;
    - 2.7 Mt of zero-grade overbreak from stoping adjacent to sedimentary rocks (based on geomechanical modelling); and
    - Inclusion of inferred KIMB3 kimberlite within the overall pipe shape as zero-grade waste.
13. Stockpile Mineral Reserves are estimated at a cut-off value of $19/t based on a rehandle cost of $2/t, a processing cost of $12/t and a G&A cost of $5/t, when processed at the end of mine life;
14. Stockpile Reserves are not included in the KDM Mineral Resource Estimate, which covered only in-situ mineralized material;
15. Stockpile Reserves are based on surveyed volumes and block model grades; and
16. Stockpile LOM diamond price is determined from the weighted average of the North, Centre, South - M/PK(s), and South - EM/PK(s) lobe ratios.

16 MINING METHODS

16.1 Introduction

KDM is an existing OP mine located in Central Botswana that has been in production since 2012. Conventional OP drill and blast mining with diesel excavators and trucks provide an average annual 2.6 Mt of kimberlite feed to the mill, plus additional ore to surface stockpiles. The OP mine operation is expected to terminate mid-2025, ending at an elevation of 713 masl. The mine currently has approximately 9.7 Mt of stockpiled reserves available for processing.

There are substantial resources remaining below the economic extents of the OP that may be extracted by UG mining methods, presented herein.

The mine design and planning for KDM is based on the resource model completed by SRK in 2019, as detailed in Section 14 of this report. The mine plan proposes the continuation of OP activities to a depth of 713 masl at which point the resource is to be mined by UG methods to a depth of 310 masl. The UG operations will provide on average 2.7 Mt/a to the processing facility and add 14 years to the mine life.

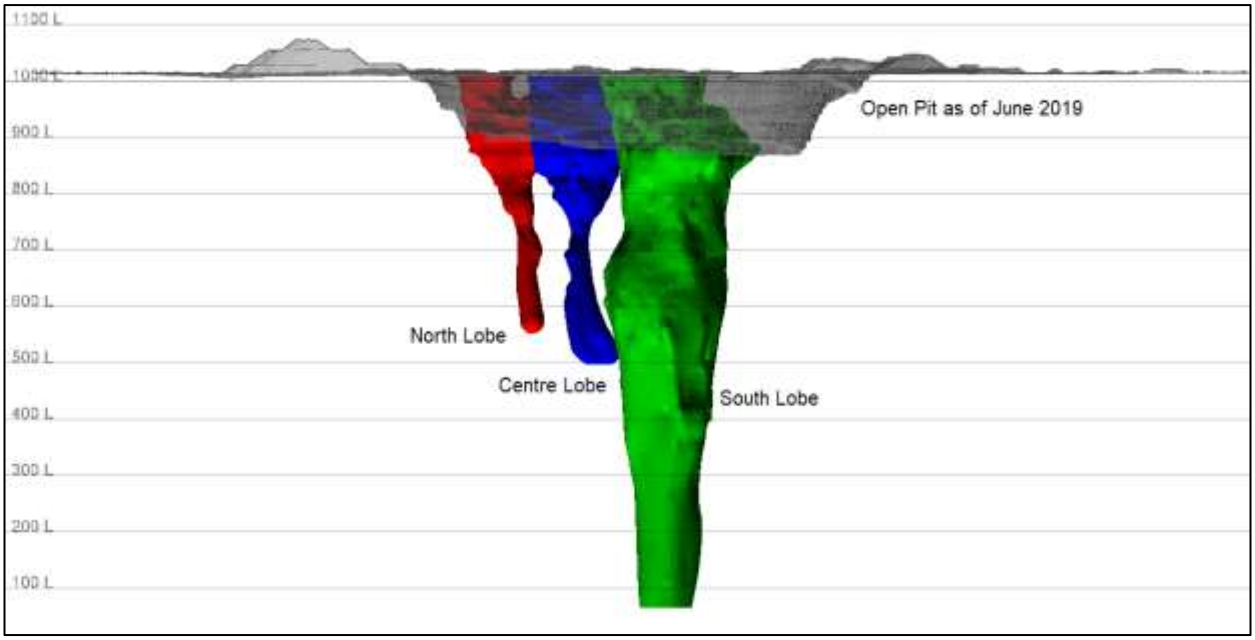
A total of 37 Mt with an average grade of 14.2 cpht will be mined from the UG operations. In 2021 construction of an UG Mine commenced which is expected to achieve steady state target production in 2028 according to a first principals, no-float, development schedule.

5.8 Mt of stockpiled OP ore will be processed during the transition from OP to UG operations, leaving 4.4 Mt of stockpile to be processed at the end of Mine Life

16.2 Deposit Characteristics

KDM resource contains three distinct coalescing pipes, referred to as the North, Centre, and South Lobes as illustrated in Figure 16-1. All lobes are outcropping, dip vertically, and vary in diameter and depth. The South Lobe is the largest of the three, and its Indicated Resources extend approximately 760 m below surface (from 1,010 masl to 250 masl). The North and Centre lobes extend below the OP limit but have been excluded from the planned UG mine as they are inferred at depth and are of low value.

Figure 16-1: North, Centre, and South Kimberlite Lobe



Source: JDS (2019)

Table 16-1 states the geometries of the South Lobe at 100 m increments.

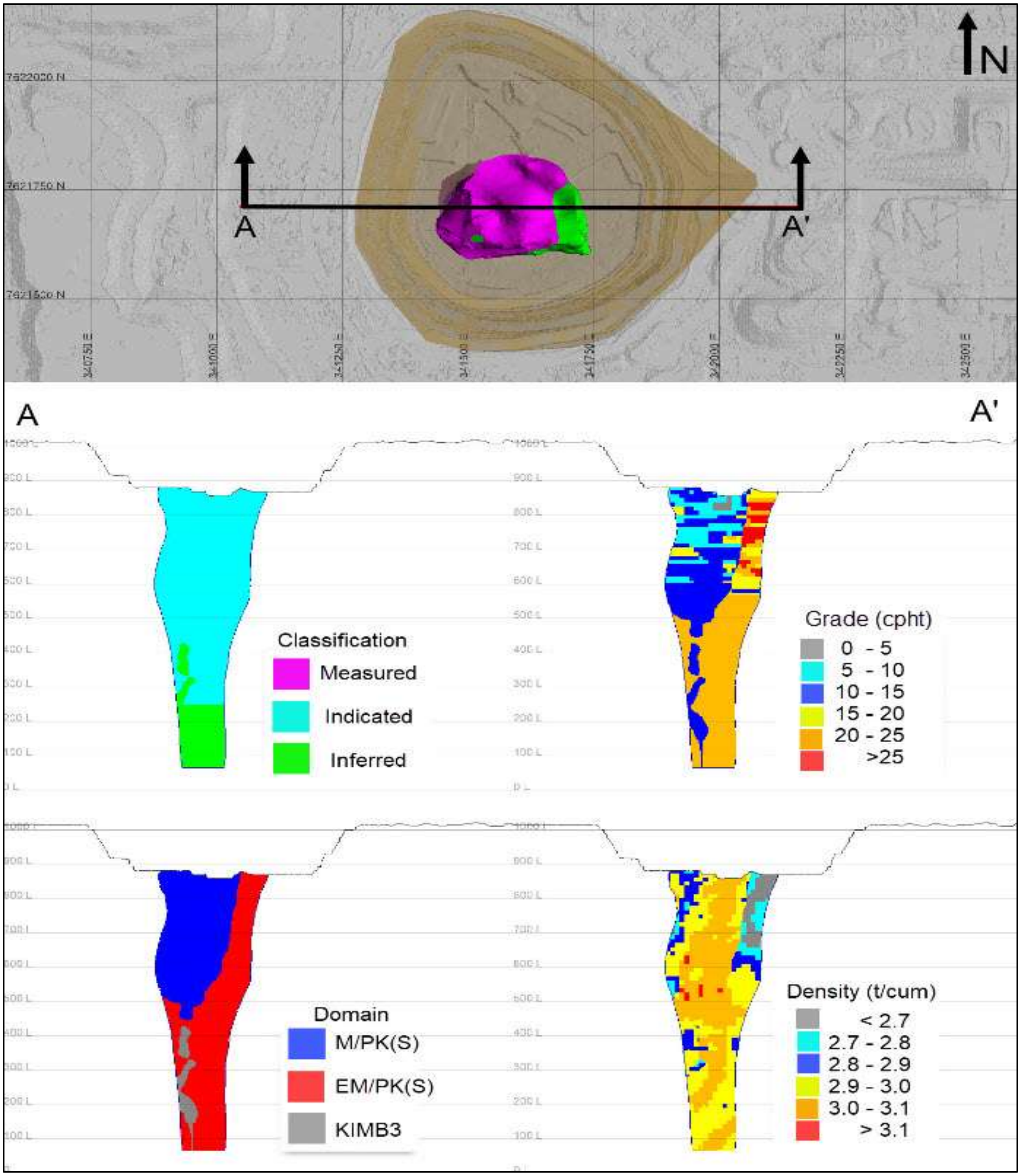
Table 16-1: South Lobe Dimensions and Hydraulic Radius

Elevation (masl)	Diameter (m)	Area (m^2)	Circumference (m)	Hydraulic Radius
800	215	36,400	703	52
700	207	33,550	668	50
600	213	35,575	704	51
500	180	25,330	592	43
400	152	18,130	528	34
300	122	11,680	389	30
200	110	9,560	355	27
100	101	8,060	325	25

Source: JDS (2019)

The South Lobe contains four distinct domains, each with unique mineral properties. These domains are discussed in greater detail in Section 6 and are summarized as EM/PK(S), M/PK(S), KIMB3, and Weathered Kimberlite. Weathered Kimberlite has been mined out by the OP and is no longer present in the Mineral Resource or reserves. KIMB3 is an inferred resource that has been, for reporting and economic modelling purposes, treated as zero-grade dilution in the UG mine plan. EM/PK(S) and M/PK(S) are the two economic mineralized domains within the South Lobe on which the UG mine plan is focused. The M/PK(S) domain is situated near surface and has approximately half the diamond grade and contained value of the EM/PK(S) domain. This geologic feature drives several mine plan design decisions which focus on accessing the deeper, higher-value EM/PK(S) resource early in the mine life. Figure 16-2 illustrates the South Lobe resources by domain, grade, classification, and density. By comparing the four figures, it becomes apparent that the deeper resources contain higher grade at a greater tonnage factor, yielding more value per cubic metre of material mined.

Figure 16-2: South Lobe Resource Cross Section Looking North



Source: JDS (2019)

16.3 Geotechnical Analysis

16.3.1 Introduction

The geotechnical aspects of feasibility assessment were addressed by the collection and analysis of new geotechnical data and analysis of the geomechanical feasibility of the candidate mining methods. The collection and analysis of geotechnical data was managed by SRK Consulting (South Africa), who provided technical advice for the setup of, quality assurance, and oversight of the geotechnical data investigation program and updating of the geotechnical model. The laboratory testing program was undertaken at an accredited testing facility, Rocklab in Pretoria, South Africa. Estimates of rock mass strength and analyses of geomechanical feasibility were provided by Itasca Consulting Group, Inc. (Minneapolis, USA) and Pierce Engineering provided technical oversight and direction. The geomechanical feasibility was reviewed by SRK Consulting (South Africa), who also provided support specifications for the sinking of the shafts, which are currently in progress.

16.3.2 Geotechnical Data Collection

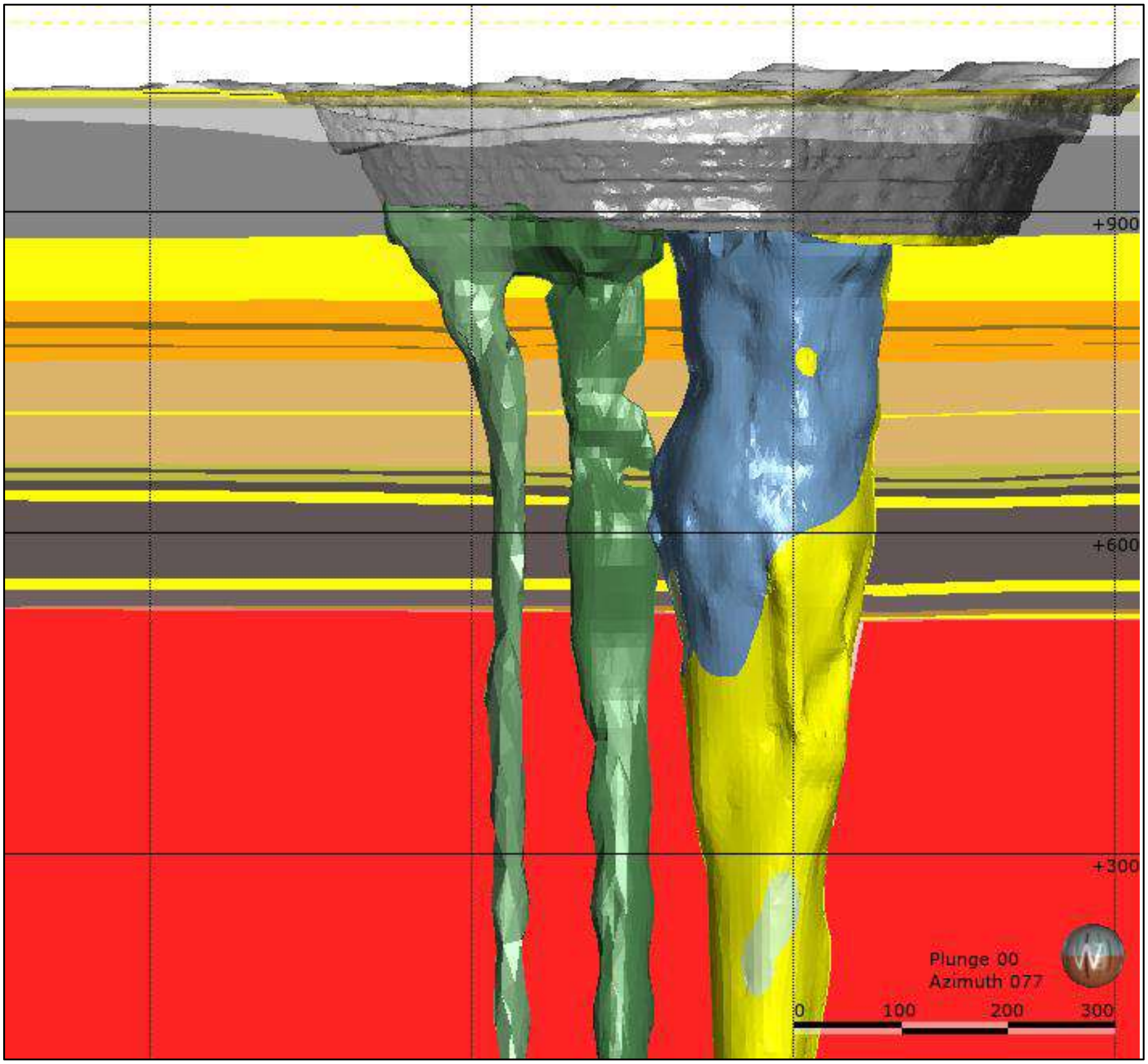
A geotechnical investigation program was carried out to support UG mine design, building on the OP and UG PEA geotechnical modelling carried out in 2017. The geotechnical drilling, sampling and testing program was designed to comply with the data confidence requirements of a FS, in support of a feasibility-level mine design, and leading into optimization of the design implementation. The investigation focused on defining the geotechnical characteristics of the surrounding country rock as well as the South Lobe kimberlite and involved the drilling, geotechnical logging and sampling of 37 diamond drillholes, totalling more than 23,500 m, with field and laboratory testing of the core samples. Acoustic Televiewer (ATV) logging was also conducted in a subset of holes to identify open joints and bedding planes and complement the oriented core logging data. Almost 11,000 tests were conducted on samples across the various lithologies, including:

- Uniaxial compressive strength tests with Young's modulus and Poisson's ratio measurements (UCM);
- Brazilian tensile strength tests (UTB);
- Triaxial compressive strength tests (TCS);
- Direct shear tests on rock joints (SHJO);
- Rock base friction angle tests (BFA);
- Rock porosity tests (POR);
- Rock Slake durability index tests (SDI); and
- Rock Duncan swelling index tests (DSI).

Key outcomes of the investigation program are as follows:

- Updating of the geological country rock, structural, and rock mass model based on the additional drilling (see Figure 16-3);
- Establishment of a detailed geotechnical logging database, including laboratory and field strength test results and structural orientation logs;
- Creation of a 3D rock mass block model that provides both statistical and spatial distributions of the project geotechnical data;
- Recording of core photographs from hyperspectral imaging program, which also provided the most reliable discernment of lithological contacts and detailed delineation of the clay content and rock mass units susceptible to weathering; and
- Mitigation of several previously identified geotechnical risks.

Figure 16-3: The Country Rock Leapfrog Model (October 2019), NNW-SSE Section Looking to ENE



Source: SRK (2019)

16.3.3 Rock Mass Quality and Strength

A summary of the rock strength and deformation tests for the different rock formations are provided in Table 16-2.

The homogenous nature of the rock units at KDM has resulted in geotechnical domains that closely follow lithology, with some additional subdomains (e.g., contact zones) established on the basis of lower intact strength. The unweathered granite basement host and south lobe kimberlite ore are both of very good quality, exhibiting high mean intact strength (UCS=137-146 MPa) and relatively sparse jointing. This, combined with its low weathering susceptibility, makes the South Lobe kimberlite atypical. Kimberlite intact strengths are lower where the kimberlite is in contact with the country rock, due to an increased clay content and jointing. The granite is also more jointed near the contact.

The bulk of the host rock above the granite, comprising approximately 345 m of sedimentary rock (shales, mudstones and sandstones of the Karoo Supergroup) and approximately 130 m of igneous rock (basalts of the Stormberg Lava Group) are of good quality, exhibiting intact strengths that are approximately half that of the granite and kimberlite (mean UCS=53-83 MPa) and similar sparse jointing.

There are some weaker layers within the country rock that exhibit low intact strengths (mean UCS=28-40 MPa). These include the upper Ntane sandstones, the red mudstone beds within the lower Mosolotsane sandstone, some layers within the Tlapana carbonaceous mudstones and the weathered granite. The Ntane sandstones are porous, with minor clays, and a high water content weakens the rock further, but does not cause degradation. The other weaker layers contain clay forming minerals and are less resistant to weathering.

Rock mass classification indicates that the formations in the area of interest have fair to good rock mass quality. The average Laubscher RMR rating is between 50 and 60 and ranges from 30 to 90. The lower kimberlite RMR values are invariably at the contact. Lower country rock values are due to localized jointing and occasionally weathering of the red mudstone and Tlapana carbonaceous mudstones and coal.

Due to the sparse jointing, it was not considered valid to estimate rock mass strength based on the Geological Strength Index (GSI) and Hoek-Brown criterion. Rock mass strength was estimated for all domains via Synthetic Rock Mass (SRM) testing instead, with inputs derived from the following parameters:

- Intact rock strength (from axial and diametral point load testing and laboratory testing);
- Basic friction angle (from axial and diametral point load testing and laboratory testing);
- Joint condition and shear strength (from geotechnical core logging and laboratory testing);
- Joint orientation and spacing (from oriented core logging and ATV logging); and
- Intact rock material constant mi (derived from laboratory test results).

The results of SRM testing suggest that large-scale rock mass UCS values are in the range of 15-39% of the lab-scale UCS (average = 26%). These strengths should be considered as representative of conditions in which the units are compressed parallel or perpendicular to bedding (where present) as point load testing revealed an intact strength anisotropy in some units. A lower tensile strength exists along surfaces parallel to bedding in the unweathered Stormberg Basalts (anisotropy index = 2.7), Ntane (anisotropy index = 1.4), Tlhabala (anisotropy index = 1.2) and Tlapana (anisotropy index = 1.2-1.9) formations. This was considered conservatively in the analysis of geomechanical performance by assuming ubiquitous horizontal bedding planes in the Ntane, Tlhabala and Tlapana units with zero tensile strength.

There are no major faults evident in the kimberlite or host sediments. A NW-SE and a WNW-ESE fracture domain was identified that shows increased subvertical fracturing. The NW-SE corridor follows the main intrusion trend of the kimberlite pipes and is accompanied by kimberlite stringers.

Table 16-2: Summary of Laboratory Strength and Deformation Characteristics

Formation	Count (UCM)	Count (TCS)	Count (UTB)	UCS (MPa)			Indirect Tensile UTB (MPa)			m_i	Elastic Modulus (MPa)			Poisson's Ratio (n)		
				Mean	CL_{lower}	CL_{upper}	Mean	CL_{lower}	CL_{upper}		Mean	CL_{lower}	CL_{upper}	Mean	CL_{lower}	CL_{upper}
Kalahari beds	44	114		83.2	67.9	98.5				15.0	35.9	30.4	41.4	0.20	0.18	0.21
Stormberg basalt - weathered	23	23	22	36.0	28.4	43.7	13.1	11.1	15.1	15.0	14.1	10.1	19.6	0.19	0.16	0.22
Stormberg basalt -unweathered	40	128	42	79.4	72.3	86.5	27.0	25.1	29.0	11.0	34.7	32.1	37.3	0.21	0.21	0.22
Ntane Formation	67	198		25.9	29.8	29.8	6.7	25.1	29.0	20.0	11.1	10.4	11.8	0.20	0.19	0.21
Mosolotsane red mudstone	20	46	19	28.4	23.5	33.2	8.1	6.4	9.8	17.0	6.6	5.0	8.7	0.17	0.15	0.20
Mosolotsane sandstone	81	229	81	47.8	43.4	52.2	11.5	10.5	12.5	19.0	16.4			0.20	0.20	0.21
Tlhabala massive mudstone	53	163	54	78.0	72.1	83.8	28.6	26.4	30.8	12.0	18.6	17.2	20.1	0.16	0.15	0.17
Tlapana SST MS CMS	8	26	9	86.3	72.1	100.5	25.9	23.5	28.3	10.0	16.8	14.4	19.1	0.17	0.15	0.18
Tlapana CMS L1	8	89	9	24.5	11.4	37.6	7.0	4.9	9.1	10.0	9.9	6.5	13.3	0.16	0.12	0.20
Tlapana CMS L2	45			45.0	34.6	50.5	13.1	10.5	16.4		10.1	8.1	12.7	0.14	0.12	0.15
Tlapana CMS L3	2	16	2	92.5	89.2	95.7	22.7	6.4	38.9	20.0	19.7	9.4	30.0	0.16	0.13	0.20
Tlapana CMS L4	4		4	95.4	46.5	144.3	24.2	9.4	39.1		19.2	16.6	21.7	0.16	0.11	0.20
Tlapana SS ARK L1	1	6	1	27.4			11.7			15.0	11.5			0.13		
TlapanaSS ARK L2	2		2	40.4	23.8	57.0	8.5	5.8	11.1		11.0	7.0	14.9	0.18	0.09	0.27
Tlapana SS SST	12	36	13	33.5	25.1	42.0	11.3	8.1	14.6	23.0	11.3	7.8	14.9	0.20	0.17	0.23
Basement granite	42	100	37	149.2	131.1	167.3	31.8	28.8	34.8	30.0	66.1	53.7	81.5	0.22	0.20	0.23
Basement granite - kaolonitised	4	27		25.0	19.3	30.5	15.5	10.0	24.0	30.0	7.5	4.1	11.0	0.15	0.12	0.18
Dyke Dolerite	5	0	8	209.3	154.1	264.6	44.3	28.0	60.6	0.0	81.5	77.2	85.8	0.27	0.25	0.29
Kimberlite North Lobe	2	0	2	70.5	49.5	91.4	15.3	5.3	25.3	0.0	20.4	17.4	23.4	0.23	0.18	0.27
Kimberlite Centre Lobe	5	10	5	119.9	83.2	156.7	28.6	25.6	31.6	14.0	44.2	36.8	51.7	0.28	0.24	0.32
Kimberlite South Lobe	32	102	35	144.7	134.0	155.4	32.8	30.0	35.6	13.0	67.2	60.5	74.0	0.24	0.23	0.26
Kimberlite South Lobe MPK	11	32	12	158.9	140.8	177.1	35.6	30.8	40.4	30.0	76.0	67.1	84.9	0.25	0.23	0.28
Kimberlite South Lobe EM/PK(S)	18	56	18	137.9	126.1	149.6	32.6	29.1	36.1	26.0	64.9	56.6	73.2	0.25	0.23	0.26
Kimberlite South Lobe EM/PK(S) K3	3	14	5	133.5	24.9	242.1	27.0	13.7	40.2	30.0	48.6	0.0	111.9	0.21	0.09	0.32

Source: SRK (2023)

16.3.4 Weathering Susceptibility

The hyperspectral imaging provided a reliable assessment of the alteration and clay content and potential for weathering. The dominant alteration and clay minerals present in the Country Rock sediments are saponite and kaolinite/calcite, with lesser amounts of illite and chlorite present, while alteration in the basalt is dominated by saponite. Kimberlite shows a general lack of alteration and clay minerals, but increased saponite and serpentinite content in the kimberlite is seen at the upper and lower kimberlite contacts, with lesser amounts of nontronite present. The basement granites are dominated by kaolinite in the upper portion (weathered/kaolinitised granite) and illite in the lower portion (unweathered granite/gneiss).

Analysis of the Mosolotsane red mudstones shows a dominance of saponite, indicating that the material expands when exposed to water and will quickly weather to a residual soil. Similar analysis of the Tlapana carbonaceous mudstones shows a dominance of montmorillonite, indicating that the materials will tend to adsorb water and disassociate, resulting in material of a dispersive nature. Both materials additionally contain varying amounts of illite and chlorite.

The kimberlite contacts show dominance of saponite and serpentinite, indicating a tendancy towards expansive clays when weathering, with lesser amounts of illite, talc and amphibole also present. The talc may be indicative of a sheared contact.

The core sampling program was designed to retain as close as possible to in-situ material conditions by wrapping and sealing weathering susceptible core immediately after exposure and sampling and packaging the core for transport to the laboratory and testing within one week after exposure. Accelerated weathering tests provided a field calibration of the durability of the weathering-susceptible materials under repeated wet-dry cycles, allowing for calibration of the laboratory test results for expected UG conditions.

The kimberlite did not demonstrate any susceptibility to weathering under wet-dry cycles due to its low clay content, but weathering may occur at the contact. The red mudstones of the Mosolotsane Formation were shown to degrade within one wet-dry cycle, while the mudstones, carbonaceous mudstones and coal layers of the Tlapana Formation exhibited a higher resistance, starting to degrade within three to five cycles. Weathering of the red Tlapana unit was observed in the televiewer logs and complete weathering of the red mudstone occurred, in the vent shaft core hole, which was drilled with double tube. The Tlhabala unit is relatively competent and has a low susceptibility in general, with only a subset of samples exhibiting degradation. As a result, the rock mass strengths estimated for the susceptible subdomains in these units should be considered representative of in-situ strengths. Exposure of these materials to atmospheric conditions (in particular water) is expected to result in a greater than 50% reduction in their rock mass strengths within a short time.

Any UG development that may take place in these materials should be sealed as soon as possible after exposure of the rock face to avoid degradation due to atmospheric exposure.

16.3.5 In-Situ Stresses

The absence of compressional features, such as faults and folds within the country rock indicates that there are not unusually high horizontal stresses in the country rock. Estimates of the magnitude and orientation of in-situ stress in the South Lobe kimberlite are based on wireline Sigra testing (overcoring method) completed by Sigra PTY Ltd. These suggest that the pipe has variable horizontal stresses, close to the vertical stress in the near-surface and higher than the vertical stress at depth.

Further stress measurements (over-coring method) should be carried out during horizontal development to access the orebody, prior to mining, to increase the confidence in the in-situ stress state.

16.3.6 Caveability

The combination of high kimberlite strength, low in-situ stresses and limited hydraulic radius of the pipe suggest that natural caving is not a viable mining approach at KDM. The variable and low horizontal stresses in the near surface would also not allow for reliable generation of horizontal hydrofractures (preconditioning). The caveability of the orebody was also examined in FLAC3D, which suggested that natural caving was not likely, tending to collapse to an arch and stabilize when undermined (does not cave continuously).

16.3.7 Pore Pressure

The evolution of pore pressures as mining progresses was also incorporated into the geomechanical model to allow for the computation of effective stresses and their associated effect on overall stability. The monthly pore pressure distributions used were received from Exigo in July of 2020 and correspond to the case where no dewatering program is employed through the UG or surface.

16.3.8 Brow and Crown Pillar Stability

Several LHS stoping sequences have been evaluated and optimized with the assistance of FLAC3D models, as different sequences lead to different levels of brow and crown pillar stability, with sequences that mimic an arched back, and employ short lead / lags and blast heights being more stable.

The Itasca Model for Advanced Strain Softening (IMASS) constitutive model was used to simulate the rock mass behavior. IMASS enables the numerical model to represent the damage around an excavation or caving process and accounts for the progressive failure and disintegration of the rock mass from an intact/jointed to a bulked material. It has been calibrated against actual stope and caving performance.

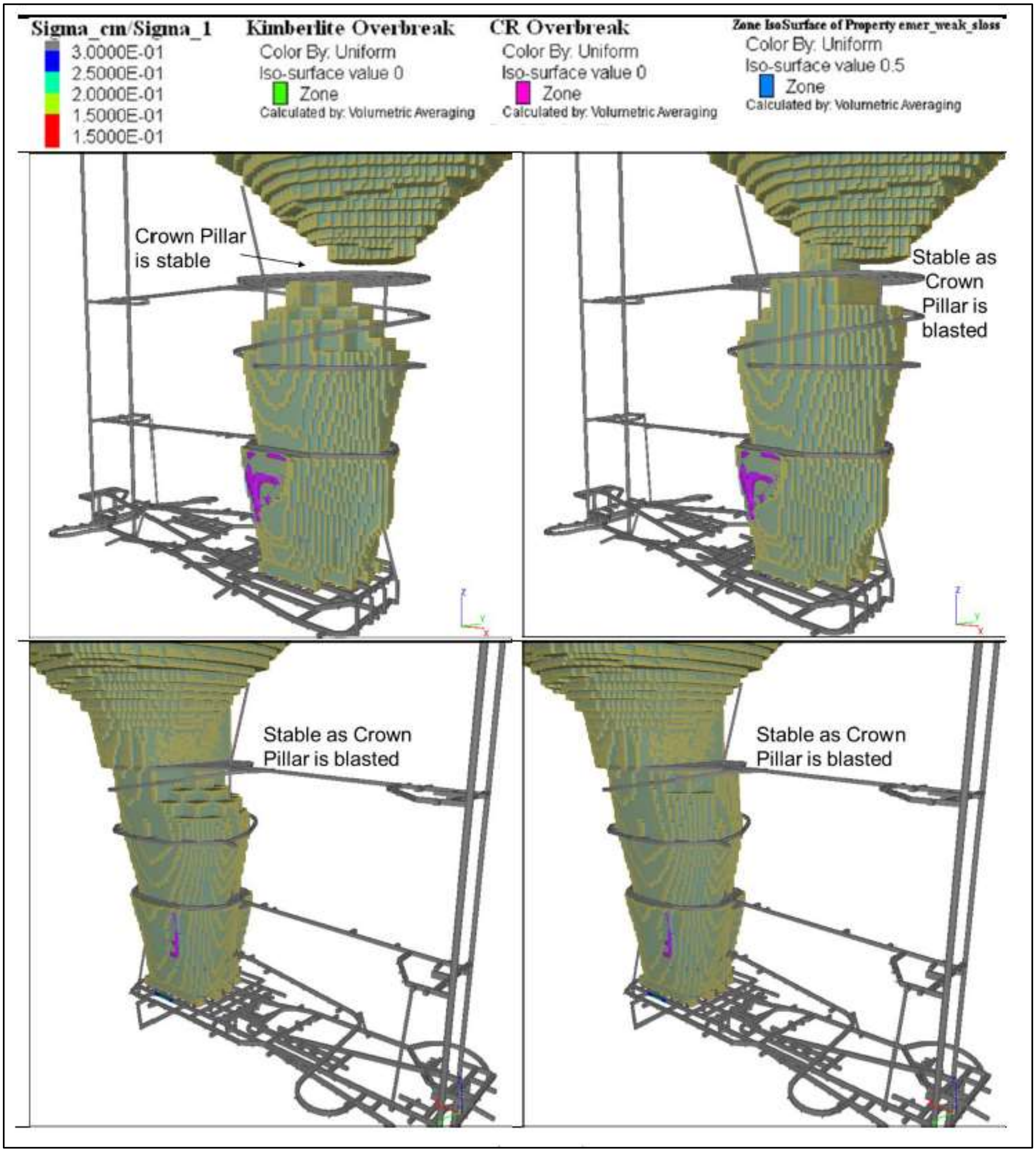
The selected pyramidal sequence has the most stable back shape, which promotes stability with low overbreak and promotes stability of the crown pillar. The results following the pyramidal LHS sequence show that the excavation back and brows are in a stable condition, with minimal likely back overbreak as stoping advances. Minimal overbreak in the country rock (Granite) local to the weathered and weakened contact between the Granite and Kimberlite pipe is predicted. Additionally, the stresses induced on the drifts and drilling horizons are not high enough to induce problematic closure strains suggesting that a 30 m vertical pillar is sufficient to ensure drill drive survivability.

The LHS sequence was advanced until a ~25 - 30 m crown pillar was formed. Figure 16-4 shows the lack of rock mass strength (damage) as the crown pillar progresses. The resulting crown pillar is stable and exhibits a factor of safety of 2.

It will be essential to inspect the blastholes with a borehole camera and measure the hole length after each blast to confirm that the overbreak is not excessive and that the drill drives are not at risk. Also, lidar cavity monitoring, accessed through drillholes will provide a reliable measure of the overbreak and stability of the brow.

The sequence for the crown pillar extraction is appropriate, but a more detailed analysis will be required at an advanced stage of mining, once the actual rock mass response has been assessed.

Figure 16-4: FLAC3D Forecast of Damage as the Crown Pillar Progresses



Source: Itasca (2019)

16.3.9 Fragmentation

The fragmentation from stope blasting is expected to be manageable, with minimal oversize, based on the blasting results achieved in the pit at similar powder factors. Some larger blocks (>2 m^3) are expected to result from natural overbreak of stope brows but will be manageable with the large number of drawpoints and planned secondary blasting capabilities. Some minor to moderate attrition of oversize is also expected from secondary fragmentation during drawdown. The results of Rapid Emulator Based on Particle Flow Code (REBOP) software simulations indicate that the percentage of fines expected at the drawpoint due to secondary fragmentation is ~10% and a reduction of oversize material in the order of 32% after drawing an equivalent 400 m height of draw.

16.3.10 Dilution Potential

FLAC3D analyses to date suggest that the potential for dilution of ore by overbreak into the surrounding country rock is very low due to the stabilizing effects of the pipe geometry (circular cross-section) but is sensitive to the assumptions around host rock in-situ stresses.

Positive pore pressures in the country rock increase the extent of yielding and will affect the predicted country rock amount that could enter as dilution. The resolution of the pore pressure data used in FLAC3D analysis does not allow for precise reflection of its impact in the vicinity of the face of a void. The adoption of pore pressure distributions that more closely follow the mining sequence could lead to reduced levels of pore pressure grading out from the excavation boundaries, which in turn could lead to lower levels of country rock predicted to fully fracture and potentially enter as dilution. While at this point in the study, the level of detail may be enough, this is something that should be revisited if and when a more detailed estimate of potential dilution is required.

The updated mine design does not include a kimberlite skin as was included in the 2019 FS and all kimberlite is planned to be extracted as the stopes progress upwards. This scenario has not been modelled, but the original analysis provides an indication of the potential dilution. The model predicts that approximately 2.67 Mt of country rock have the potential to enter as dilution.

In practice, dilution must be managed through draw control. Only the swell should be extracted to provide space for the next blast, to ensure that the host rock remains confined. Once the crown pillar has been extracted, the draw rate can be increased, but must still remain uniform to prevent waste dilution from entering the muckpile.

Cavity monitoring through drillholes and ultimately from surface will be important to provide a 3D survey of potential overbreak and subsidence.

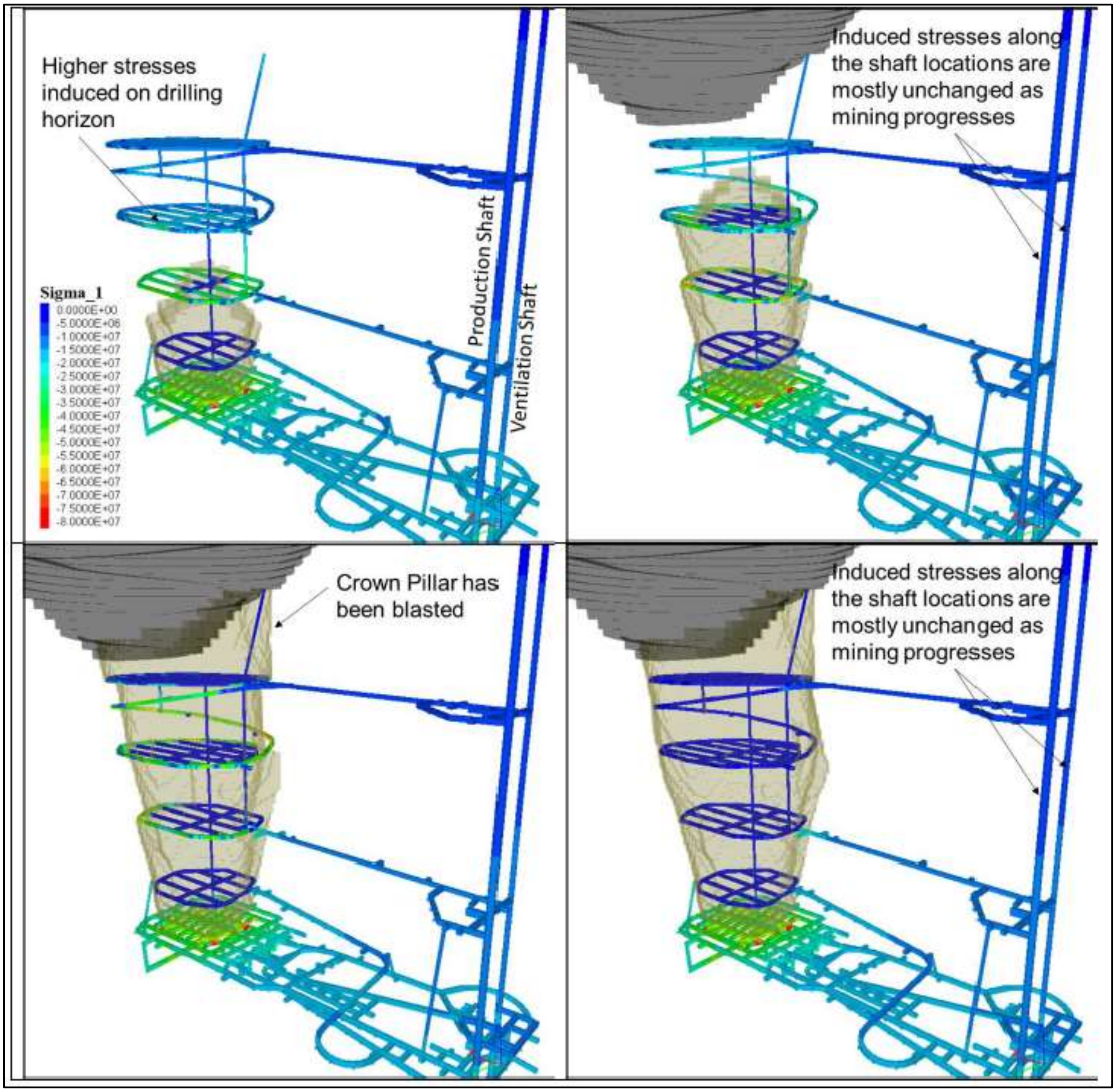
16.3.11 Infrastructure Stability

Vertical and lateral development in the kimberlite and much of the host rock encountered is expected to be very stable due to the sparse open and low to moderate induced stresses. Empirical support design methods will be adequate as a result. The exception is where weathering susceptible units (see Section 16.3.4) are encountered in the shaft, where special care should be taken to seal and support these exposures.

With the pyramidal LHS sequence selected, drill drives are predicted to be stable as the stope back approaches (inducing higher stresses) and a 25 m sill pillar is recommended to ensure drill drive survivability (FOS > 1.3). FLAC3D analysis of induced stresses suggests that haulage drifts should be placed >15 m away from footprint to minimize induced stress changes and closure strains.

Figure 16-5 shows the induced stresses painted on the infrastructure as a result of blasting the Kimberlite pipe (semitransparent brown volume), showing that the elevated Sigma1 induced stresses (most compressive principal stress) are only very local to the pipe and do not extend to the vicinity of the shafts. The modelled strains in the shaft indicate that they will be within tolerable limits.

Figure 16-5: Sigma1 Stresses (Most Compressive Principal Stress) Painted on Infrastructure as LHS Mining Progresses, the Semi-Transparent Brown Volume Represents the Blasted Kimberlite



Source: Itasca (2021)

Sinking of the P/S and V/S has commenced, and no significant stability problems have been experienced. The support design caters for weak layers and potential weathering (weathered basalt, red mudstone, Tlapana carbonaceous mudstones, and weathered granite). A special support and sinking sequence were designed specifically for the red mudstone. A permanent, full-shaft concrete lining, 0.3 m thick will provide long term stability, in addition to the temporary support.

16.3.12 Subsidence Potential

The FLAC3D model predicts that no damaging surface subsidence is expected prior to crown pillar blasting. The potential for damaging subsidence to occur beyond the final pit crest after the crown pillar is blasted is considered low based on analyses to date but should be re-examined when a higher resolution of pore pressures is available for inclusion in the FLAC3D mechanical analyses of host rock stability.

During mining it will be important to monitor the potential sloughing of the country rock from the rim tunnels and access drives on 380L, 480L, 580L, and 680L. If additional resolution is required, monitoring drillholes will be necessary. It is essential verify the expected rock mass behaviour and update the FLAC3D analyses as mining progresses.

16.3.13 Hazards

The potential for mud rush is considered to be low given the high strength, low clay content and low weathering susceptibility of the kimberlite. Good draw control will limit the ingress of clay minerals into the muckpile during the mining sequence. Subsequent sloughing of country rock will fall on top of the muck pile and will only become a potential concern at the end of the life, when the muckpile height is very low. Good draw control will remain essential throughout the life of the mine.

There is a low risk of seismicity due to the relatively low stress:strength ratios expected around development. This should be confirmed through early stress measurements (overcoring method) from access tunnels, prior to commencement of mining.

The risk of air blast is to be managed by minimizing the height of the air gap during upward advance of the shrinkage stopes and by blasting the crown pillar before substantial drawdown occurs.

The pyramidal mining sequence creates a compressive arch, which will clamp blocks, until the extraction of perimeter stopes in the sill, when this effect is reduced. Geological features (not included in the model) may allow the formation of large blocks, which could topple into the excavation. This risk is increased during the extraction of the perimeter stopes in the sill, when blocks will be bounded by free faces, the weaker, jointed contact zone, and possible faults. Sudden or unplanned block failures could result in equipment and personnel falling into the stope. Structural and geotechnical mapping of the drill horizon development, and the contact zone, followed by the preparation of a structural model, will assist in the evaluation of the potential for block failure.

The stope back shape, rock condition and broken muckpile level will be continuously monitored with geotechnical devices like extensometers and cavity monitoring systems. Mass blasting of perimeter stopes may be required to ensure safety of personnel. Perimeter drives in the host rock will mitigate the risk associated with the perimeter stopes in the sill. This will allow more escape routes and the perimeter stopes could be blasted through additional blastholes drilled from the host rock perimeter drive. There will be an additional cost due to the additional waste development, and recovery and fragmentation may be compromised.

The current crown pillar extraction sequence is complex, incorporating a mass blast, which is necessary for the safety. There is a risk that there are unusual geotechnical conditions, or the rock mass response is different from that anticipated. If the risk of crown pillar failure cannot be mitigated, this may cause resource sterilization. During mining, the rock mass response will be monitored and assessed. As more information is gathered on the geotechnical characteristics and behaviour, it will be necessary to update the model to take this into consideration. The stability of the crown pillar should be re-assessed and re-designed if required.

16.4 Hydrogeology Analysis

16.4.1 Introduction

Since the release of the 2019 feasibility study report, five key updates were made:

- The groundwater flow model was updated with the MINEDW code instead of FEFLOW code;
- The UG drainage gallery at the 680 Level (680 L) that was planned to be installed in January 2021 in the 2019 feasibility study (FS) was not implemented;
- The groundwater flow model in the 2019 FS assumed that grouting in the granites will take place in all UG development and will be 75% successful. The predicted inflow rate in the updated model (2023) only assumed 66% successful grouting during shaft sinking and station development up to January 1, 2026; the model also assumes that no grouting activities are undertaken once UG pumping capacity is available unless particularly high inflows are encountered that hinder development;
- The UG drainage systems were updated; and
- The depressurization target for the OP slope was updated.

These key updates, along with basic hydrogeologic information and updated groundwater flow model predictions, are presented in this section.

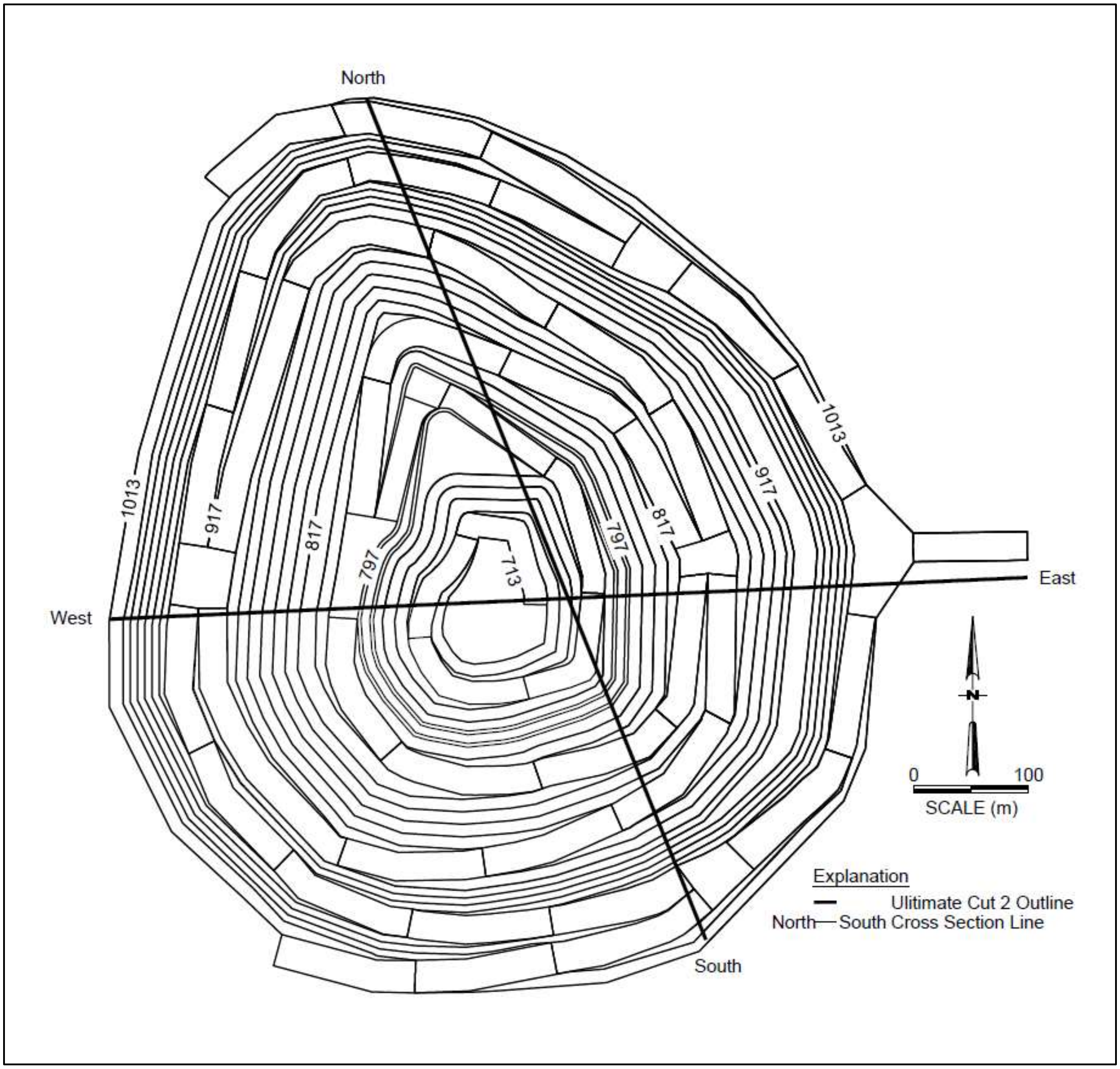
16.4.2 Mine Planning and Scheduling

The OP mining will be completed in May 2025. Figure 16-6 shows the plan view of the OP. Figure 16-7 and Figure 16-8 show the current and final pit shells along with geology settings along east-west and north-south cross sections. The following observations can be made from these two figures:

- The current pit bottom elevation is approximately 796 masl and is within the Mosolotsane unit; and

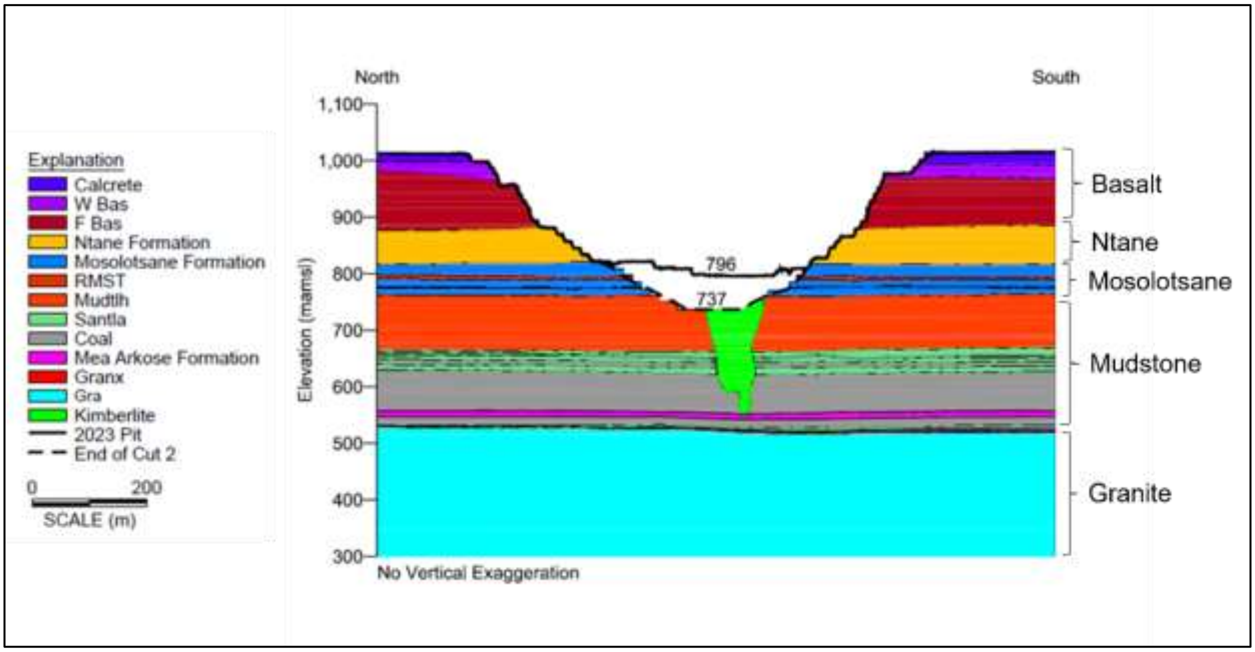
- The final pit bottom elevation is approximately 713 masl and is within the Tlhapala mudstone unit and kimberlite.

Figure 16-6: Footprint of Ultimate Pit and Locations of Section Lines



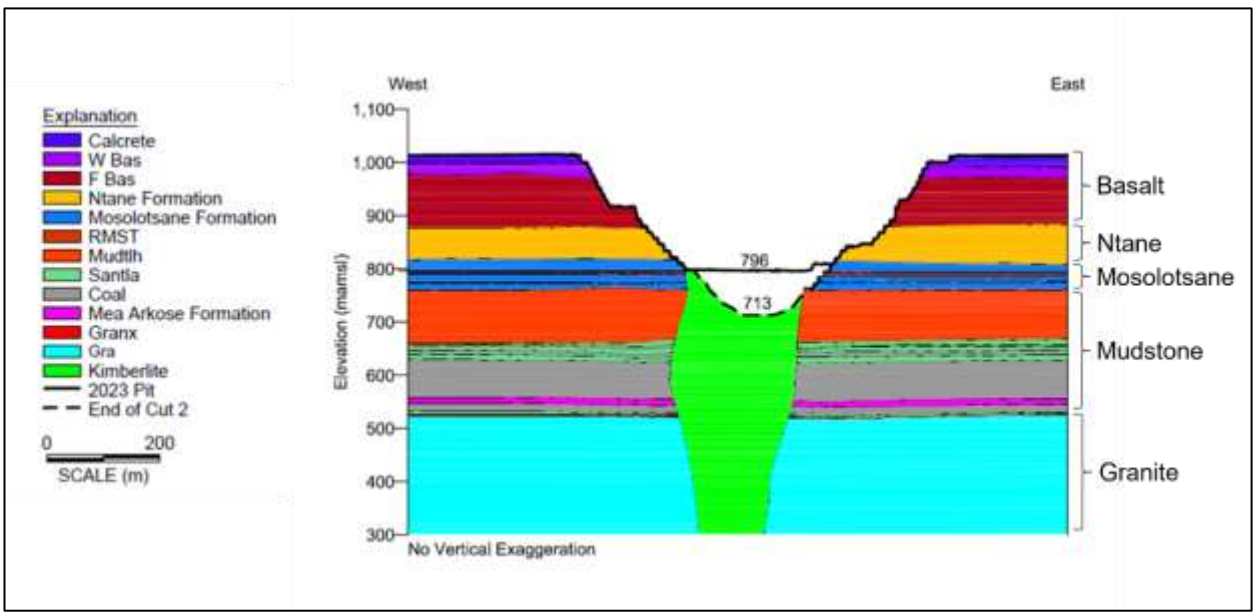
Source: Lucara (2023)

Figure 16-7: North-South Cross Section



Source: Itasca (2023)

Figure 16-8: East-West Cross Section



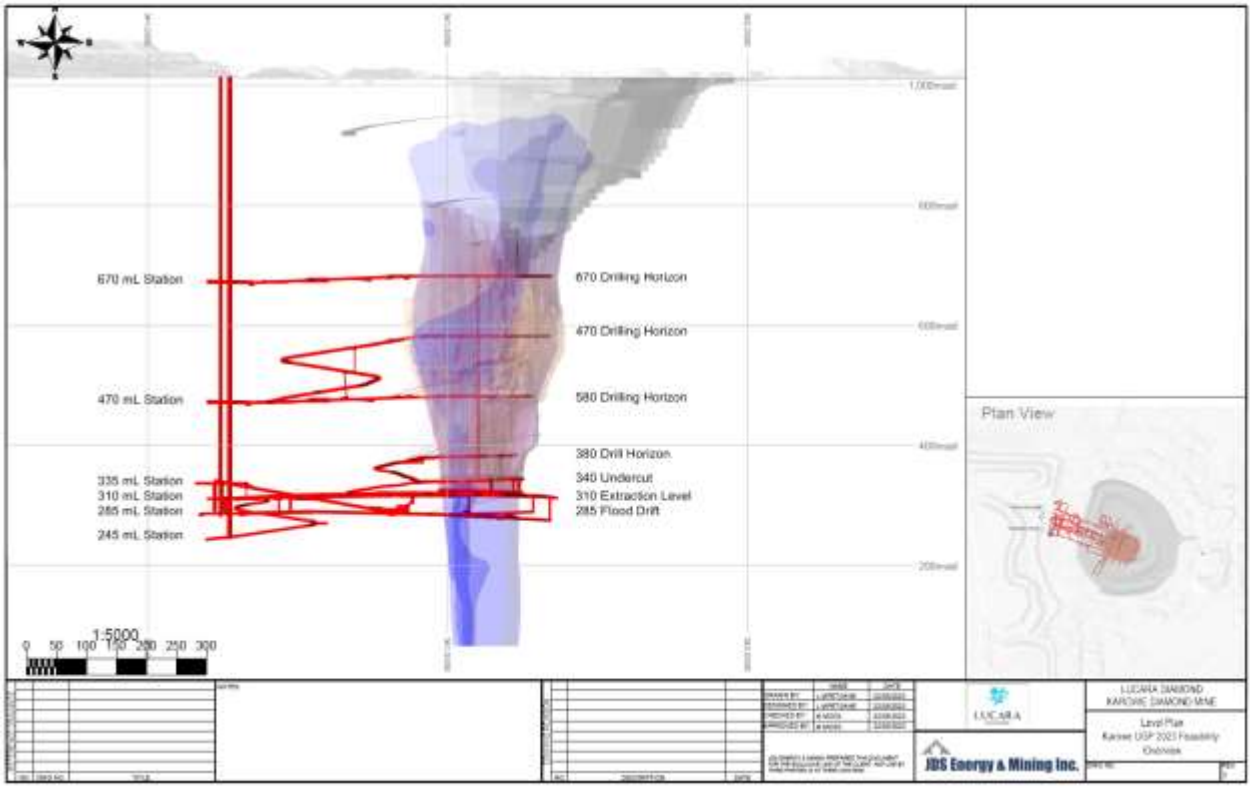
Source: Itasca (2023)

The updated UG mine plans are shown in Figure 16-9 and Figure 16-10. The mining schedule is provided in other sections of this report.

Figure 16-9: Plan View of the UG Mine Layout

Source: JDS (2023)

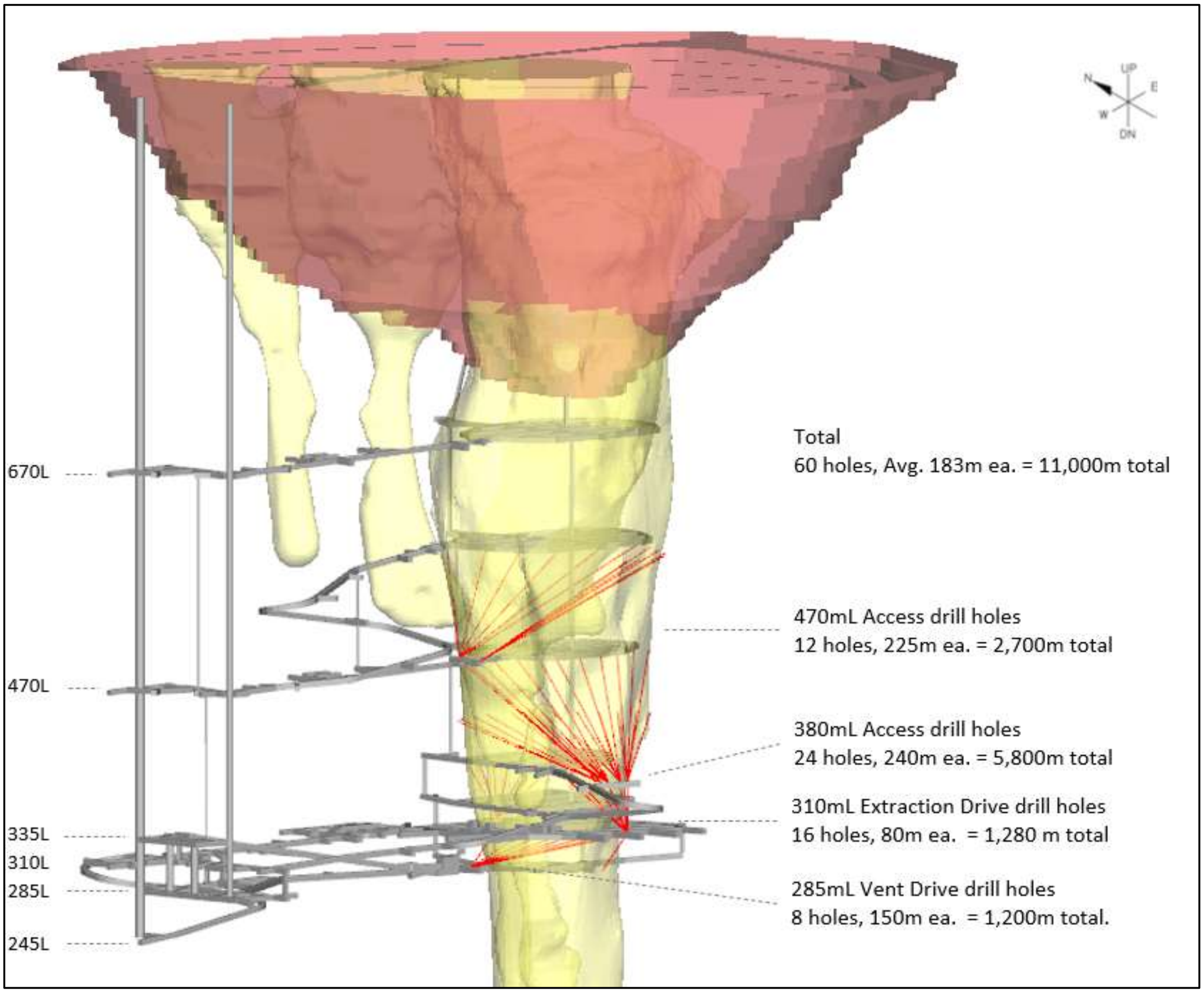
Figure 16-10: Section View of the UG Mine Layout along Section A-A'



Source: JDS (2023)

The dewatering and drainage system for the UG mining is shown in Figure 16-11. A total of 60 drain holes will be drilled at four different mine levels to dewater the mining areas. The average length of the holes is 183 m.

Figure 16-11: UG Drainage Gallery in 2023 Mine Design

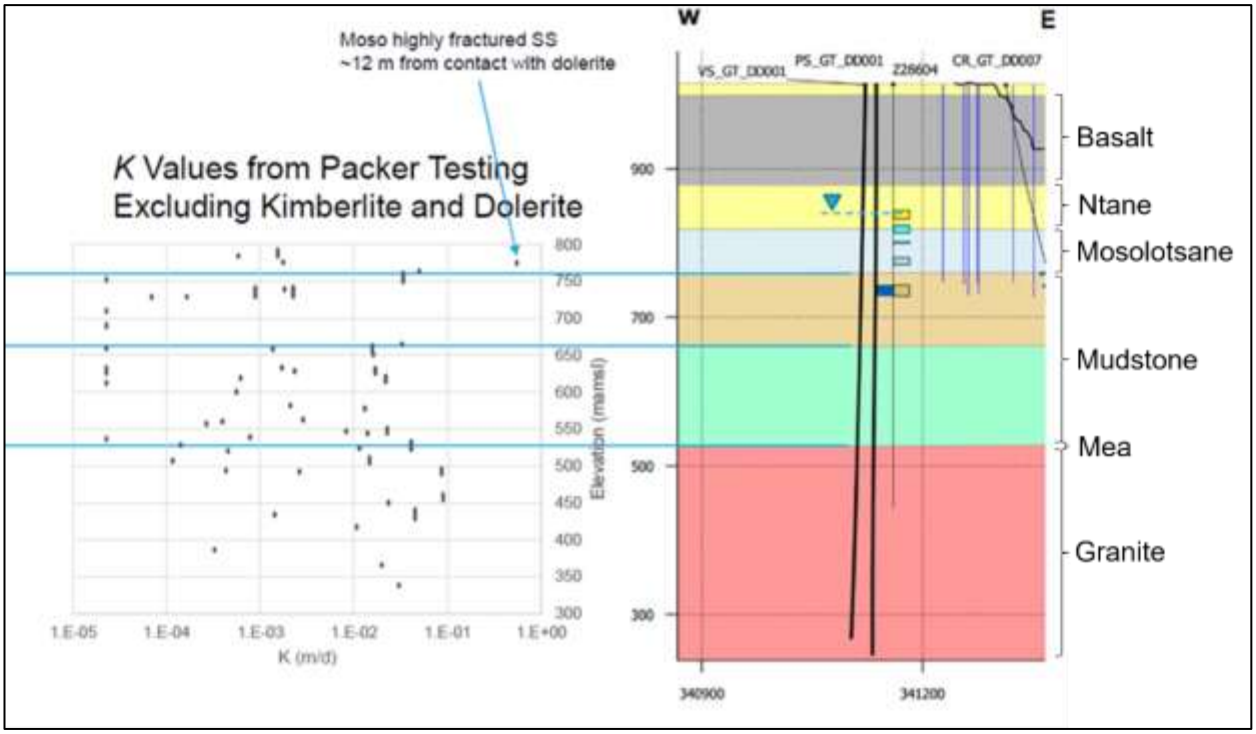


Source: JDS (2023)

16.4.3 Hydrogeologic Data Review, Gathering, and Analysis

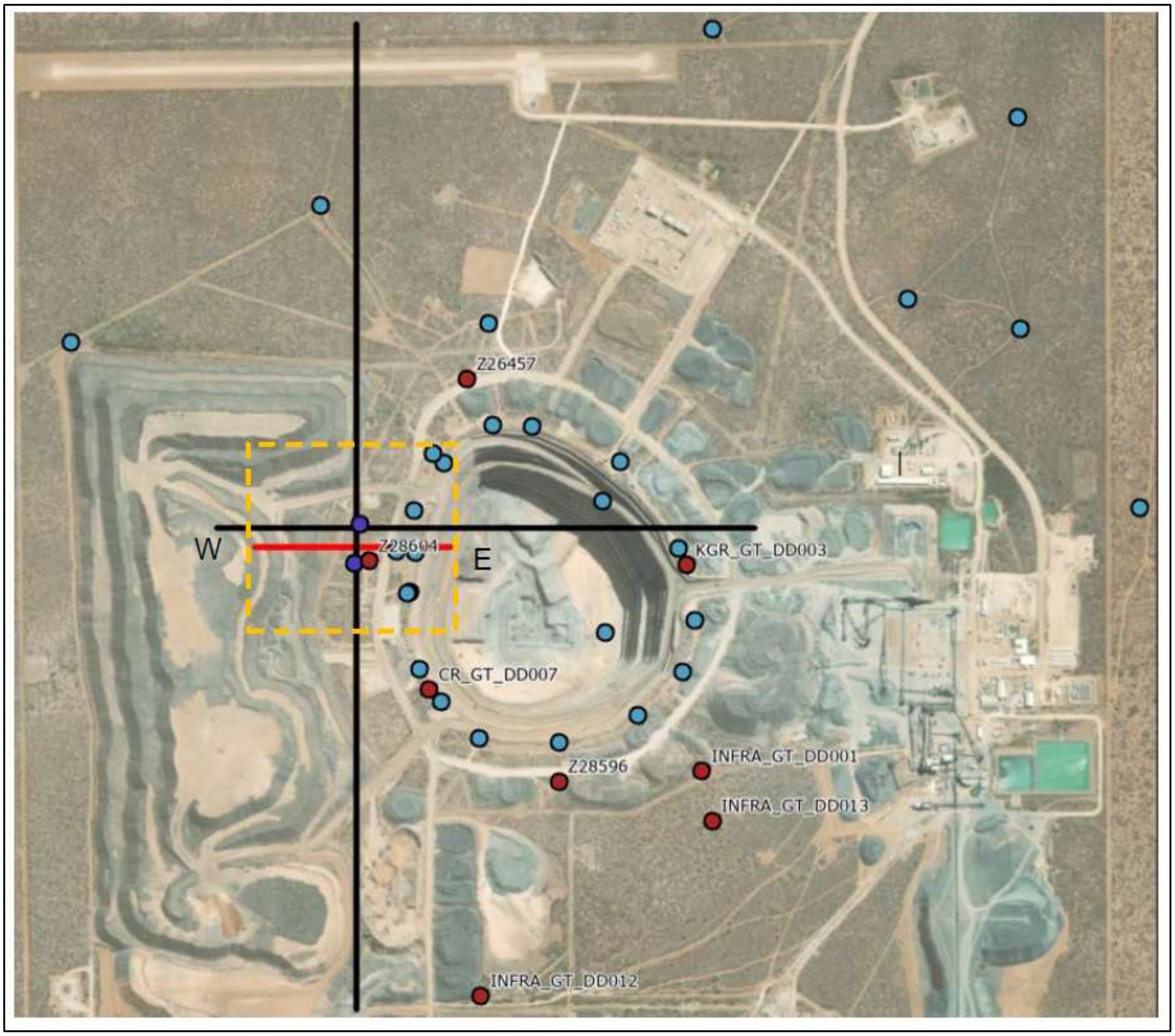
No additional hydrogeologic investigations were conducted to gather additional hydrogeologic parameters since 2019. The measured hydraulic conductivity (K) values along the depth from the past packer testing are summarized in Figure 16-12; the location of the cross section in is shown in Figure 16-13.

Figure 16-12: Distribution of Measured Hydraulic Conductivity along Depth from Packer Testing



Source: Itacsa (2022)

Figure 16-13: Locations of Boreholes and W-E Section Line



Source: Itacsa (2022)

The geologic setting is typical in the region and similar to those at the Orapa and Letlhakane OP. Ntane and Mosolotsane are regional sandstone with relatively higher *K* values than the mudstone units. The *K* values of the sandstone are mostly less than 0.1 m/day, which is considered to be low-permeability porous media. The OP operation is currently within the Mosolotsane unit and confirms that the sandstone units are low-permeability groundwater units. The majority of the measured *K* values in the mudstone unit are below 0.01 m/day, and it can be considered a very low- to low-permeability geologic unit. The measured *K* values of the Mea and granite units mostly range from 0.01 to 0.1 m/day, and they are considered to be low-permeability geologic units, however, the Mea is quite variable in terms of water flows and permeability and/or fracture water inflows could be considerably higher in certain discrete areas.

16.4.4 Groundwater Management of OP Operation

16.4.4.1 Pit Sump

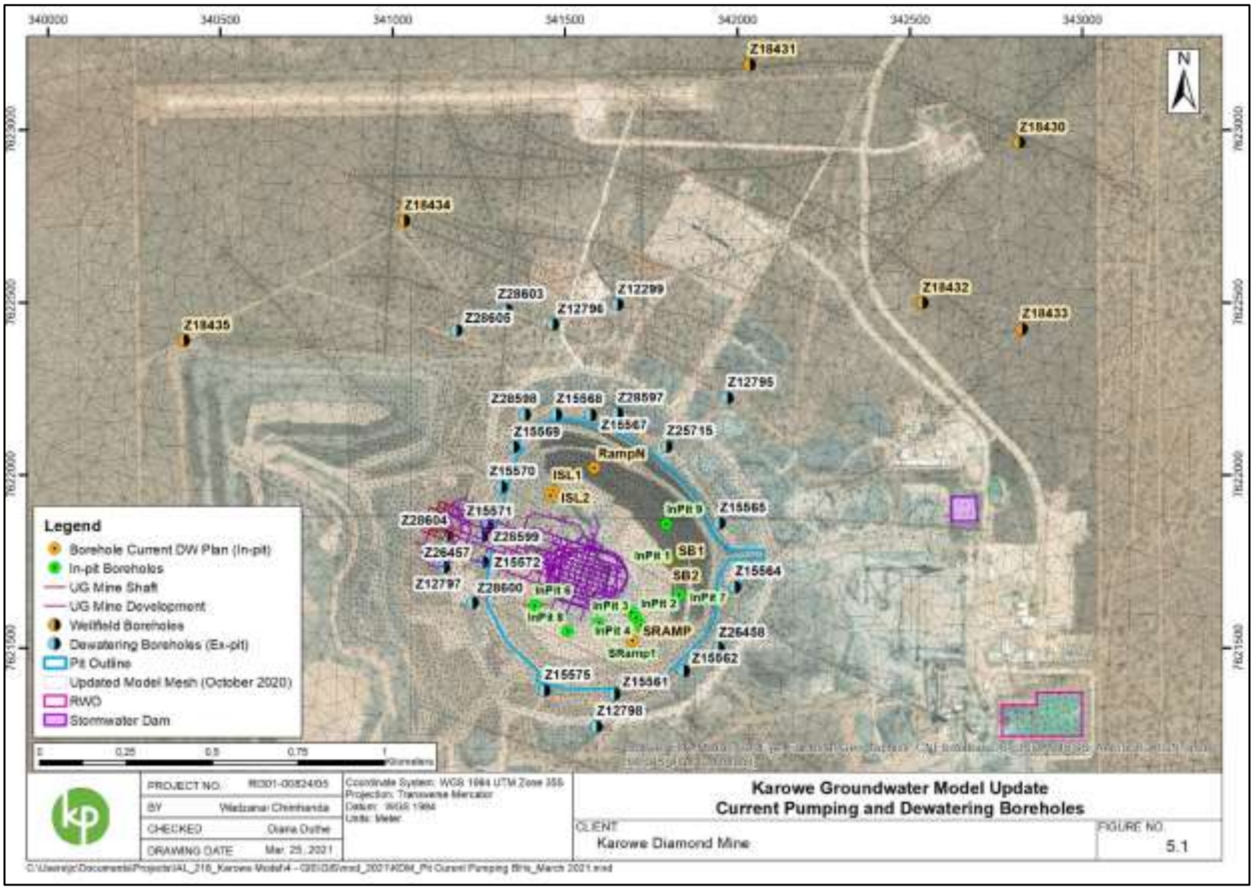
A sump pump has been used to manage the water in the pit sump. Between January 2021 and June 2023, the pumping rate from the sump ranged from 0 to 4,000 m^3/day. The average pumping rate is 500 m^3/day, and the standard deviation is 580 m^3/day. The high pump rate from the sump is the result of the runoff from precipitation during the wet season.

16.4.4.2 Dewatering Boreholes

Figure 16-14 shows the active dewatering boreholes for the OP operation. Also shown in green color in Figure 16-14 are the recommended dewatering boreholes simulated in KP (2021) for the life of the OP operation. All these boreholes are for the dewatering and depressurization of the OP operation. No active dewatering boreholes from the surface are planned for the dewatering of the UG mine. There are 25 pit perimeter dewatering boreholes pumping at approximately 200 m^3/hr in total and five in-pit dewatering boreholes pumping over 35 m^3/hr in total.

At the time of this report preparation, the implementation of the future dewatering boreholes has not been finalized. Therefore, the recommended dewatering boreholes in KP (2021) were assumed to be implemented in the prediction of the inflow to the UG mine workings over the LOM. Because the active dewatering boreholes and UG mine workings are separated by approximately 200 m thick low-permeability mudstone, the variation of pumping rates of future dewatering boreholes from those in KP (2021) will have a minor effect on the predicted inflow rate to the mine workings, which is demonstrated in a later section of this report.

Figure 16-14: Existing and Recommended Dewatering Boreholes



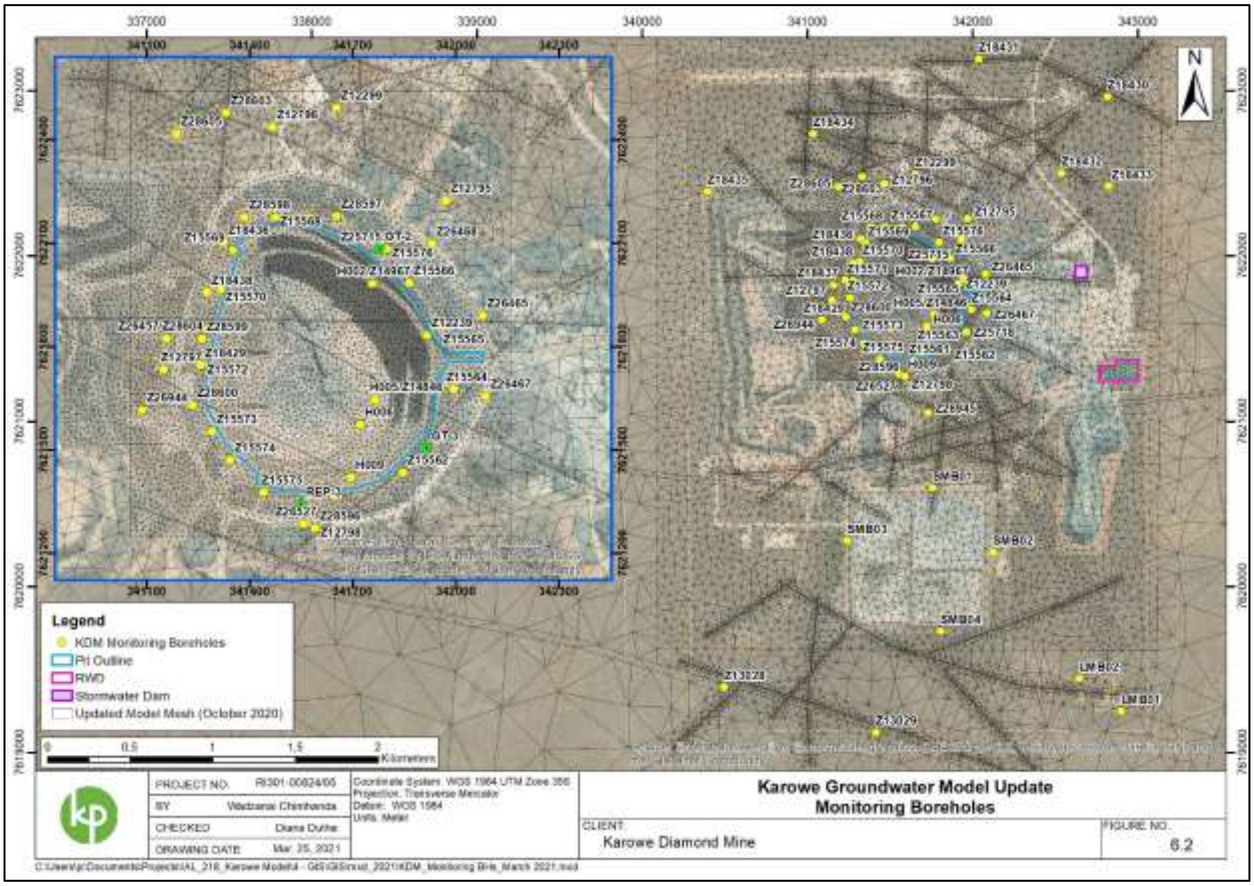
Source: KP (2021)

16.4.5 Groundwater Monitoring and Groundwater Levels

The groundwater levels have been monitored through monitoring boreholes as shown in Figure 16-15. Almost all monitoring boreholes are located in sandstone units. There are no monitoring boreholes or piezometers in the Mea and granite units.

Most of the pit-perimeter dewatering boreholes show that the measured groundwater levels range from approximately 845 to 860 masl and remain relatively stable. There are no measured water levels in the Mea and granite units. However, based on the field observation of artesian flow during the drilling at the shaft area which intersected the Mea and terminated in the granite unit, it appears that groundwater heads in the Mea and granite could be higher than the ground surface.

Figure 16-15: Existing Monitoring Boreholes



Source: KP (2021)

16.4.6 Hydrogeochemistry and Mine Water Quality

The natural baseline water quality from the regional Stormberg Basalt-Ntane contact water strike has a total dissolved solids (TDS) signature of 1,500 mg/L to 2,000 mg/L. The deep granites have saline water with 25,000 to 33,000 mg/L TDS. Water quality results from ongoing monitoring is summarized in more detail in Section 18.2.

16.4.7 Mine Dewatering Modelling and Piezometric Pressure

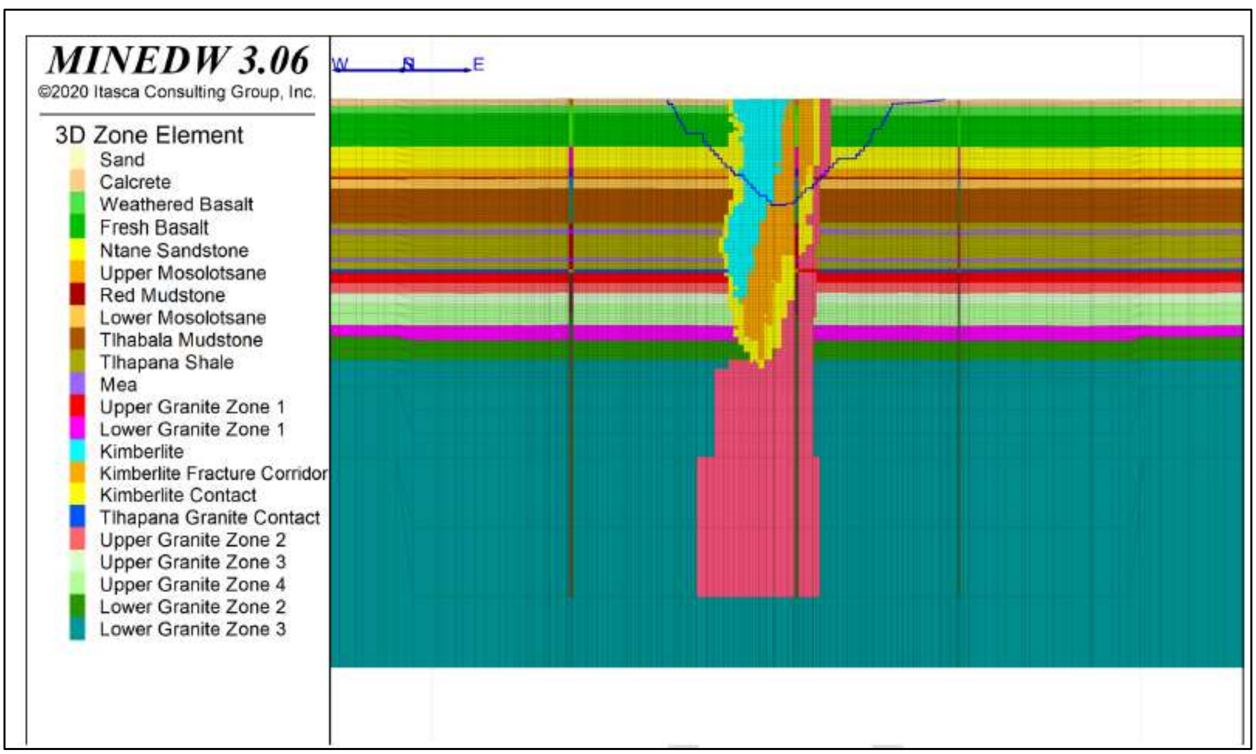
16.4.7.1 Description of Groundwater Flow Model

The groundwater flow model using MINEDW (Itasca 2012) was initially developed by Itasca South Africa for depressurization analysis and pore pressure simulations for slope stability analysis (Itasca South Africa 2020). The model was last updated by KP in 2021 (2021 Model) and used to predict the inflow rate to the UG mining. The objectives of KP's model update included the following:

- Hydrogeologic data collection and analyses;
- Model update and assessment of dewatering performance of existing dewatering systems;
- Provision of pore pressure and phreatic surface for stability analyses;
- Use of the model to optimize the dewatering strategy to ensure dewatering and depressurization targets are met within the required timeframe;
- Provision of inflow estimates into the UG workings; and
- Development of a dewatering strategy for the UG workings.

The 2021 Model was reasonably calibrated and was used for the prediction of groundwater inflow to the UG mine workings and pore pressure distributions. The simulation of the geologic units in the 2021 Model is shown in Figure 16-16.

Figure 16-16: Simulated Geologic Units in 2021 Groundwater Flow Model



Source: KP (2021)

The calibrated hydraulic parameters from the model calibration were summarized in Table 16-3. The hydraulic parameters used in the model are within the range of the measured *K* values shown in Figure 16-17.

Table 16-3: Simulated Hydraulic Parameters in 2021 Groundwater Flow Model

Model Unit	Hydrostratigraphic Unit	LITH CODE: SRK	Horizontal Hydraulic Conductivity, $K_{x,y}$ (m/d)	Vertical Hydraulic Conductivity, K_z (m/d)	Specific Storage, S_s (1/m)	Specific Yield, S_Y (-)
SAND	Sand		1.00E-01	1.00E-02	1.00E-04	5.00E-02
SAND_FC	Sand Fracture Corridor		1.00E-01	1.00E-02	1.00E-04	5.00E-02
CALC	Calcrete		1.00E-01	1.00E-02	1.00E-04	1.00E-03
CALC_FC	Calcrete Fracture Corridor		2.50E-01	2.50E-01	1.00E-04	1.00E-03
WBSLT	Weathered Basalt		5.00E-02	1.00E-02	2.00E-06	1.00E-03
WBSLT_FC	Weathered Basalt Fracture Corridor		7.50E-02	7.50E-02	2.00E-06	1.00E-03
FBSLT	Fresh Basalt		1.00E-02	1.00E-03	1.00E-06	1.00E-03
FBSLT_FC	Fresh Basalt Fracture Corridor		2.50E-02	2.50E-02	1.00E-06	1.00E-03
NTANE	Ntane Sandstone		1.50E-01	1.50E-02	5.00E-06	5.00E-02
NTANE_FC	Ntane Sandstone Fracture Corridor		3.50E-01	3.50E-01	5.00E-06	5.00E-02
UPMOS	Upper Mosolotsane		2.00E-02	2.00E-02	5.00E-06	1.00E-02
UPMOS_FC	Upper Mosolotsane Fracture Corridor		4.00E-02	4.00E-02	5.00E-06	1.00E-02
REDMUD	Red Mudstone	WI MOSO L3	1.00E-03	1.00E-04	1.00E-06	1.00E-03
L_MOS	Lower Mosolotsane		3.50E-02	3.50E-03	2.00E-06	1.00E-03
L_MOS_FC	Lower Mosolotsane Fracture Corridor		7.00E-02	7.00E-02	2.00E-06	1.00E-03
TLA	Tlhabala Mudstone		1.00E-04	1.00E-04	1.00E-06	1.00E-03
TLA_FC	Tlhabala Mudstone Fracture Corridor		1.00E-03	1.00E-03	1.00E-05	1.00E-03
TLP	Tlhapana Shale		5.00E-04	5.00E-04	1.00E-05	1.00E-03
TLP_FC	Tlhapana Shale Fracture Corridor		7.50E-04	7.50E-04	1.00E-05	1.00E-03
MEA	Mea	SS/ARC Tlapa L1, L2, L3	2.00E-02	2.00E-02	5.00E-06	1.00E-02
MEA_FC	Mea Fracture Corridor	SS/ARC Tlapa L1, L2, L3	4.00E-02	4.00E-02	5.00E-06	1.00E-02
UGRAN_z1	Upper Granite Zone 1		5.00E-02	5.00E-03	5.00E-05	1.00E-03
UGRAN_z2	Upper Granite Zone 2		1.00E-02	1.00E-03	5.00E-05	1.00E-03
UGRAN_z3	Upper Granite Zone 3		5.00E-03	5.00E-03	1.00E-06	1.00E-03
UGRAN_z4	Upper Granite Zone 4		1.00E-03	1.00E-03	1.00E-06	1.00E-03
UGRAN_FC	Upper Granite Fracture Corridor		7.50E-02	7.50E-02	5.00E-05	1.00E-03
LGRAN_z1	Lower Granite Zone 1		5.00E-04	5.00E-04	1.00E-06	1.00E-03
LGRAN_z2	Lower Granite Zone 2		2.00E-04	2.00E-04	1.00E-06	1.00E-03
LGRAN_z3	Lower Granite Zone 3		1.00E-04	1.00E-04	1.00E-06	1.00E-03
LGRAN_FC	Lower Granite Fracture Corridor		7.50E-03	7.50E-03	5.00E-05	1.00E-03
KIMB	Kimberlite		1.00E-03	1.00E-04	1.00E-06	1.00E-03
KIMB_FC	Kimberlite Fracture Corridor		2.00E-03	2.00E-04	1.00E-06	1.00E-03
KIMB_CNT	Kimberlite Contact		1.00E-02	5.00E-02	2.00E-06	1.00E-03
TLP_GRA	Tlhapana Granite Contact		2.00E-02	2.00E-03	5.00E-06	5.00E-03
DYKE	Dyke		1.00E-03	1.00E-03	1.00E-06	1.00E-03

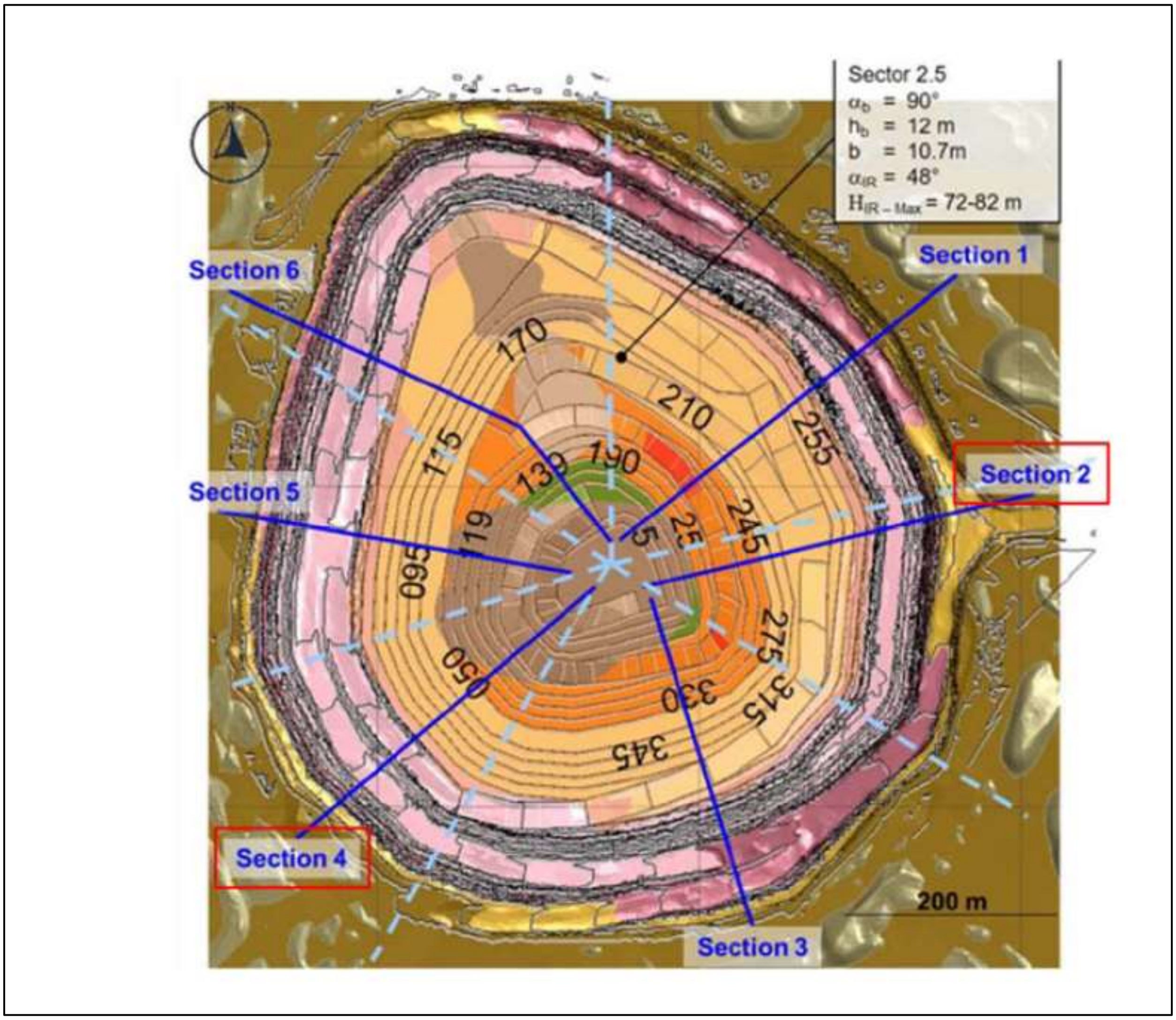
Source: KP (2021)

One key assumption in the model is the inclusion of a fracture corridor, which was based on the limited data from the previous investigation as summarized in the 2019 FS report.

16.4.7.2 Pore Pressure Target for Slope Stability Requirement

Since 2019, slope stability analyses have been conducted to assess the slope stability of the pit shell and target phreatic surface. The most recent analyses were conducted for the Cut 2 DB11 pit plan by Itasca in 2022 (Itasca 2022). In its analysis, Itasca selected the two sections that have the lowest factor of safety (FoS) that was previously identified by SRK (2020). These two sections are Section 2 and Section 4, as shown in Figure 16-17.

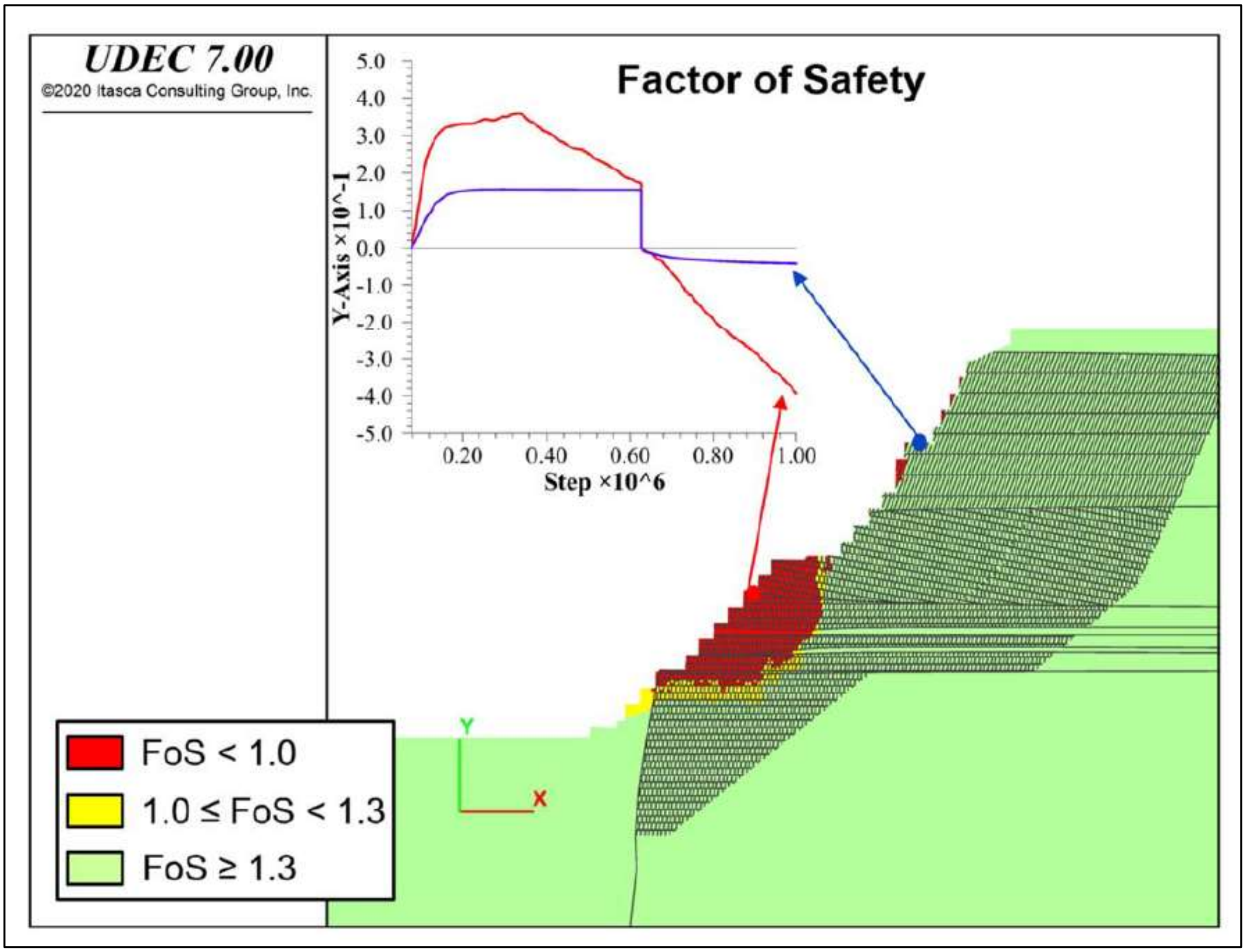
Figure 16-17: Design Sections for Slope Stability Analysis



Source: Itacsa (2022)

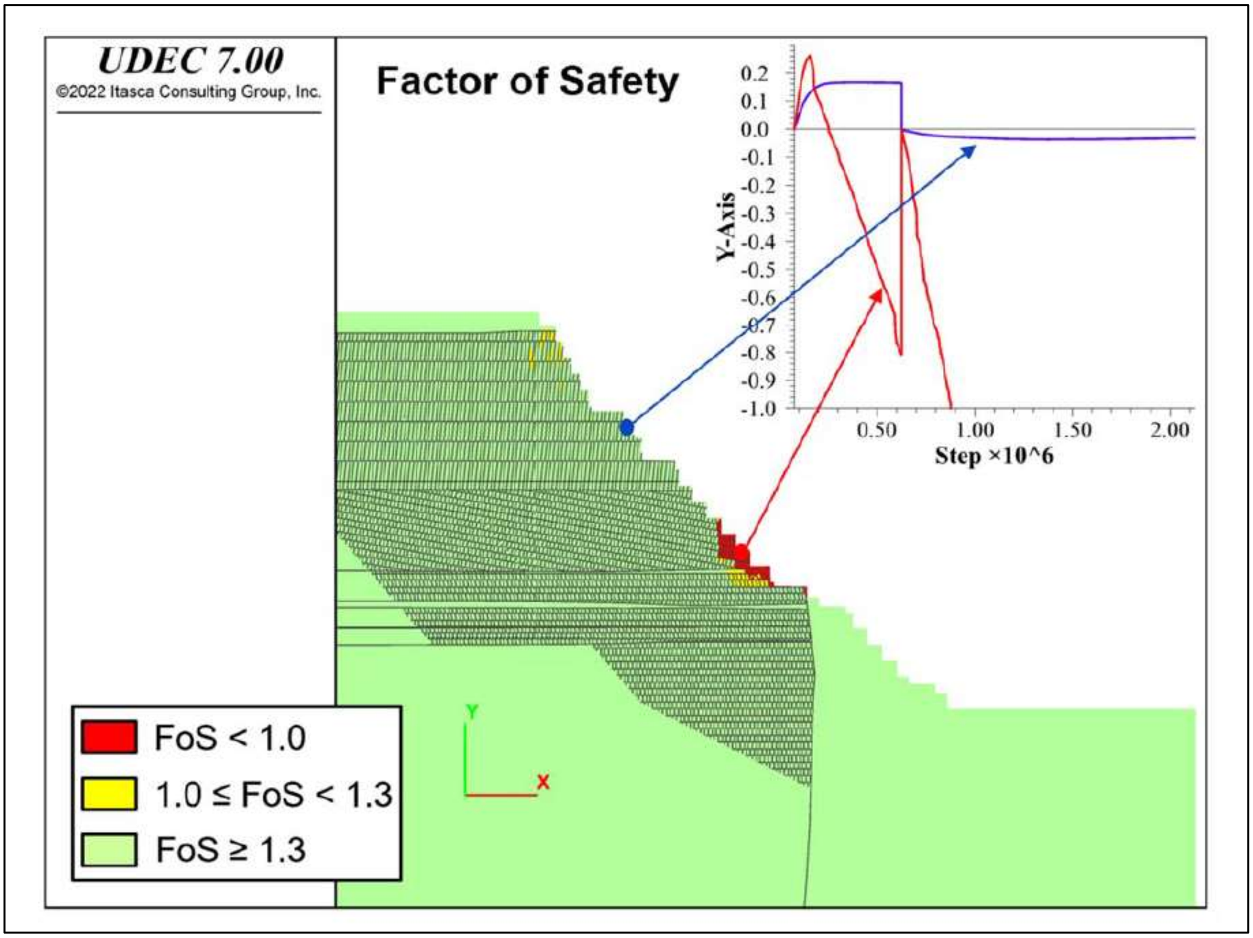
By assuming that the pit slope is fully saturated below 850 masl, Itasca found that segments of Sections 2 and 4 will have FoS values lower than 1.0. The locations of these unstable segments are shown in Figure 16-18 and Figure 16-19 for Sections 2 and 4, respectively.

Figure 16-18: Distribution of FoS of Section 2 under Fully Saturated Condition



Source: Itacsa (2022)

Figure 16-19: Distribution of FoS of Section 4 under Fully Saturated Condition

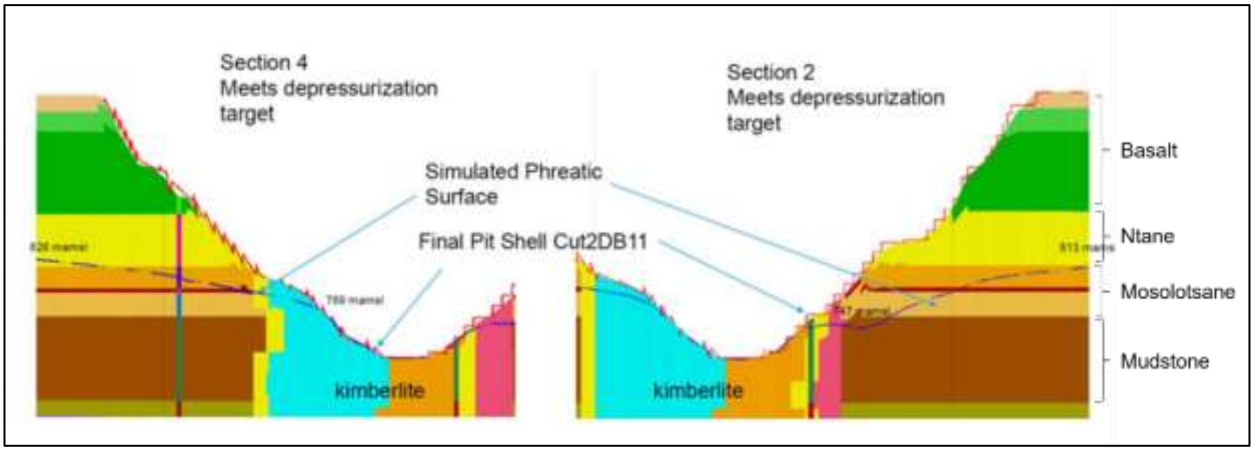


Source: Itacsa (2022)

Based on Itasca's analysis, the target phreatic surface should be 10 to 20 m behind the pit slope in order to achieve the FoS being equal to or greater than 1.3 (Itasca 2022). In order to meet the depressurization target, Itasca recommended the following dewatering plan based on the 2021 Model simulation (Figure 16-20). In that model simulation, 10 in-pit dewatering boreholes were assumed to have been implemented in April and May of 2021. The additional depressurization drain holes should be implemented as follows:

- Six by May 2022;
- 14 by December 2023; and
- 15 by 2025.

Figure 16-20: Simulated Phreatic Surface at the End of OP in December 2025



Source: Itacsa (2022)

16.4.7.3 Predicted Inflow Rate to UG Mining

16.4.7.3.1 Key Parameters and Scenarios of Model Predictions

The 2021 Model was used to predict the inflow rate to the UG operation over the LOM. Since 2021, there has been no update to the geology model; therefore, the model was only updated with the dewatering rates, OP mine plan, UG mine plan, and UG drainage gallery.

Ungrouted exploration boreholes that could potentially be intercepted by UG mining for the mine plan prior to the 2023 updated version were identified, as shown in Figure 16-21 and Figure 16-22. The coordinates and lengths of these ungrouted boreholes are summarized in Table 16-4.

The following are the key parameters used in the predictive simulations:

<u>Dewatering Boreholes</u>

All existing dewatering boreholes and recommended dewatering boreholes as presented in Figure 16-23. The details of these existing dewatering boreholes and seven recommended dewatering boreholes are provided in KP (2021) and referred to as Scenario 3. All existing dewatering boreholes were updated with actual dewatering rates up to June 2023. The future dewatering rates for the existing dewatering boreholes were assumed to continue at the most recent dewatering rates until they become dry. For the recommended dewatering boreholes, the initial dewatering rates were assumed to range between 10 and 15 m^3/hr.

<u>OP</u>

The 2023 updated OP mine plan was incorporated.

UG Mining

The 2023 updated UG mine plan and drainage gallery were incorporated.

Ungrouted Exploration Boreholes

There are multiple iterations in the predictive simulations, and predicted results were provided to JDS for different purposes, as described in the following two scenarios:

- Base Case Scenario: This scenario provides the input for UG water management purposes with the following key parameters:
  - 10 ungrouted exploration boreholes were excluded in the predictive simulations because seven of them will not be intercepted with the updated layout of the mine workings (green color in Table 16-4) and three of them will be grouted prior to their interception by the mine workings (yellow color in Table 16-4); and
  - A grouting effectiveness of 66% was incorporated for all UG developments until January 1, 2026, with a target maximum flow rate of 1,000 m^3/day to the mine workings.
- Scenario 1: This scenario provides the input for water balance purposes with the following variation from the Base Case Scenario:
  - All ungrouted exploration boreholes in Table 16-4 were simulated in the model; and
  - Grouting in the Base Case Scenario was not simulated.

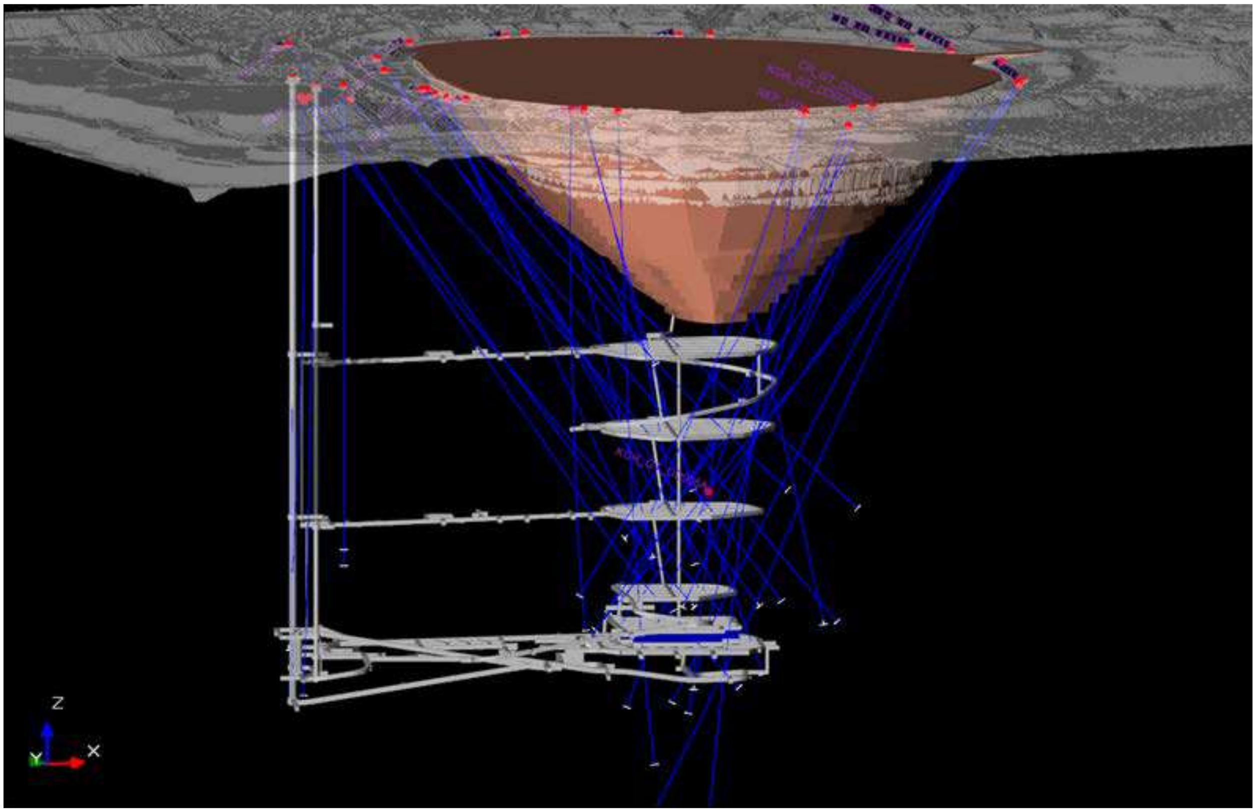
Effect of Future Dewatering on the Inflow Rate to the UG Workings

Because of the uncertainty of the planned future dewatering, another model scenario, designated as Scenario 2, was conducted to assess the effect of future dewatering on the predicted inflow rate.

Other Parameters and Assumptions:

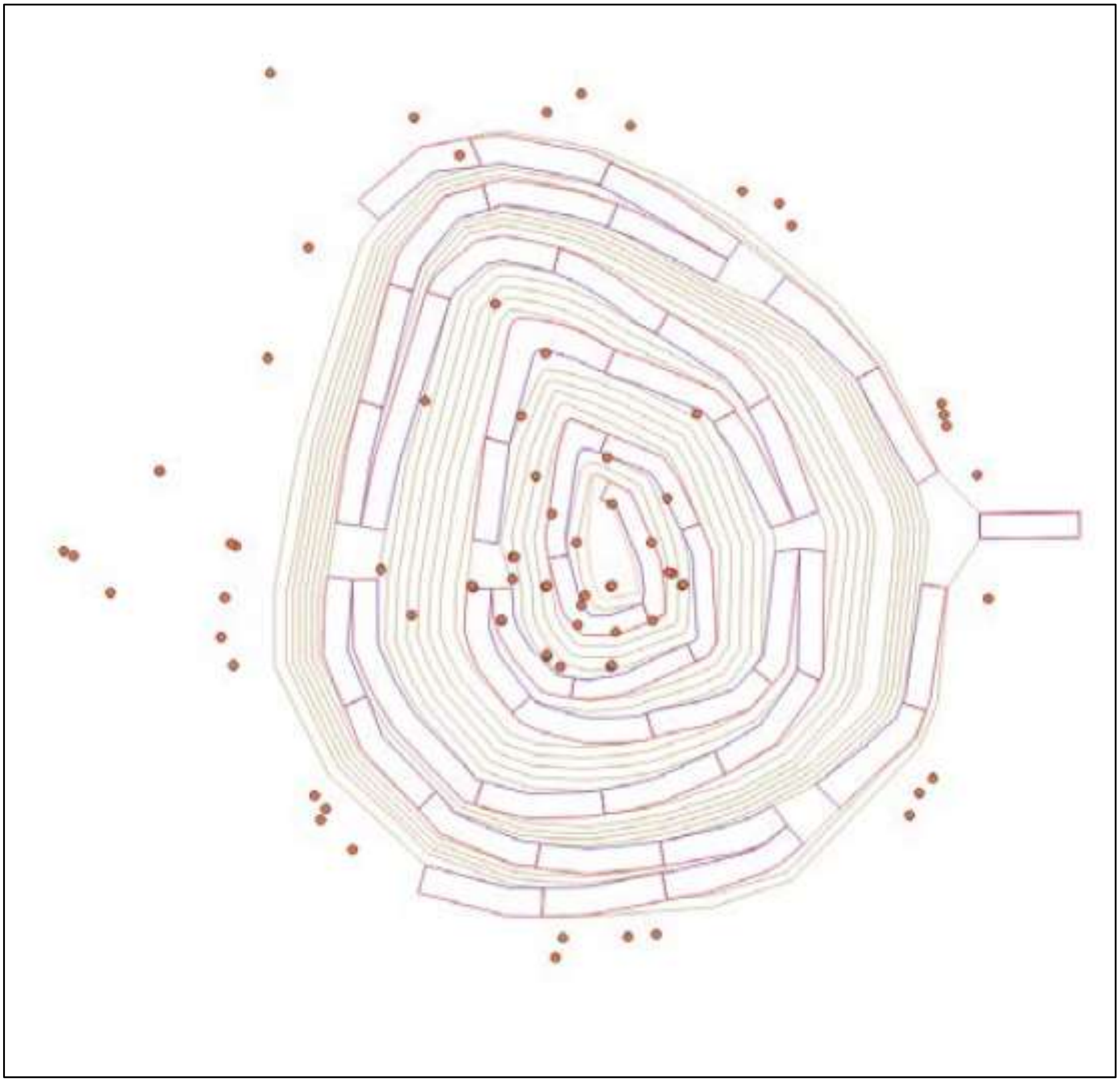
- The shaft will be 100% grouted;
- There is no active dewatering from the ground surface using active dewatering boreholes to dewater the UG mine;
- All UG workings remain open over the life of mine;
- All planned UG dewatering galleries are simulated;
- The surface-water runoff during rainfall events that will flow to the UG workings is not included in the predicted inflow rate; and
- Except for the grouting requirements prior to January 1, 2026, no grouting was planned and simulated in the model over the life of mine.

Figure 16-21: Locations of Ungrouted Exploration Boreholes Potentially Intercepted by Planned Mining



Source: JDS (2023)

Figure 16-22: Plan View of Ungrouted Exploration Boreholes Potentially Intercepted by Planned Mining



Source: Lucara (2023)

Table 16-4: Summary of Ungrouted Exploration Boreholes

Borehole ID	Top UTM Coordinates			Bottom UTM Coordinates		
	X (mE)	Y (mN)	Elevation (masl)	X (mE)	Y (mN)	Elevation (masl)
CR_GT_DD001	341266.5	7621935.8	1013.0	341773.4	7621716.8	333.7
CR_GT_DD003	341740.5	7622102.7	1012.3	341647.7	7621486.4	363.2
CR_GT_DD004	341943.8	7621868.7	1011.9	341468.1	7621509.8	391.5
CR_GT_DD005	341929.8	7621517.0	1012.9	341428.6	7621676.6	346.6
CR_GT_DD006	341654.9	7621361.3	1014.0	341416.3	7621695.5	387.0
CR_GT_DD007	341313.9	7621500.5	1012.6	341500.9	7621857.1	321.3
CR_GT_DD008	341220.7	7621658.1	1014.6	341599.3	7621816.4	344.6
CR_GT_DD010	341545.2	7622181.8	1011.8	341650.5	7621667.7	281.2
DDH002	341583.9	7621700.2	1012.7	341583.9	7621700.2	562.6
DDH004	341670.2	7621722.3	1012.3	341518.8	7621595.3	663.0
DDH006	341667.2	7621722.7	1012.2	341440.5	7621765.8	583.3
DDH009	341579.6	7621689.9	1012.7	341425.3	7621653.0	756.3
DDH010A	341543.8	7621941.6	1011.8	341604.1	7621669.9	537.0
DDH011	341519.9	7621879.4	1012.2	341632.4	7621859.5	705.9
DDH012	341510.6	7621715.4	1012.4	341685.8	7621739.4	710.5
DDH013	341510.5	7621715.3	1012.4	341648.0	7621624.6	724.2
DDH015	341694.8	7621881.7	1011.9	341452.7	7621769.4	561.0
DDH016	341494.4	7621989.9	1011.7	341539.1	7621623.0	475.6
DDH018	341605.3	7621837.4	1012.1	341671.1	7621660.3	632.1
DDH020	341458.5	7622139.1	1011.9	341594.5	7621697.2	258.7
DDH023	341665.4	7621796.9	1012.2	341700.5	7621680.1	738.1
DDH030	341534.5	7621819.1	1012.2	341445.3	7621681.6	539.5
DDH032	341380.0	7621725.0	1011.2	341656.1	7621655.7	523.2
DDH038	341410.5	7621680.1	1011.6	341556.4	7621618.7	655.2
DDH047	341423.7	7621893.4	1011.9	341497.9	7621750.6	721.2
GT01a	341319.2	7621475.8	1013.4	341613.0	7621782.4	405.4
INFRA_GT_DD003	341560.9	7621357.4	1014.2	341610.3	7621751.7	21.9
INFRA_GT_DD004	341351.6	7621446.5	1014.5	341527.4	7621715.2	171.2
INFRA_GT_DD007	341547.9	7621202.8	1014.1	341594.1	7621762.6	225.4
INFRA_GT_DD008	341985.3	7621695.5	1013.0	341505.1	7621697.6	93.6
KGR_GT_DD001	341412.7	7622176.9	1012.2	341578.2	7621781.9	461.9
KGR_GT_DD002	341789.1	7622069.2	1012.2	341525.2	7621613.4	487.5
KGR_GT_DD003	341974.3	7621819.5	1012.7	341413.7	7621668.0	330.4

Borehole ID	Top UTM Coordinates			Bottom UTM Coordinates		
	X (mE)	Y (mN)	Elevation (masl)	X (mE)	Y (mN)	Elevation (masl)
KGR_GT_DD004	341906.9	7621480.0	1013.5	341478.9	7621737.6	327.9
KGR_GT_DD005	341626.7	7621359.5	1014.6	341554.3	7621650.0	477.6
KGR_GT_DD005A	341559.0	7621629.0	515.1	341528.3	7621798.0	232.2
KGR_GT_DD006	341324.2	7621486.6	1013.5	341639.4	7621844.8	487.8
KGR_GT_DD007	341224.0	7621696.8	1014.0	341807.7	7621731.4	469.4
KGR_GT_DD008	341307.7	7622047.0	1013.0	341643.9	7621663.0	365.9
KGR_GT_DD011	341613.5	7621663.5	869.0	341528.4	7621718.0	273.2
LDD006	341574.9	7621752.8	1012.3	341574.9	7621752.8	654.3
LDD009	341576.5	7621670.3	1012.9	341576.5	7621670.3	657.9
LDD015	341650.1	7621752.4	1012.2	341650.1	7621752.4	604.2
LDD016	341650.3	7621673.5	1012.7	341650.3	7621673.5	604.7
LDD017	341499.5	7621673.9	1012.2	341499.5	7621673.9	604.2
LDD018	341511.4	7621737.6	1012.3	341511.4	7621737.6	604.3
LDD023	341680.1	7621709.5	1012.9	341680.1	7621709.5	562.9
LDD024	341545.0	7621638.6	1013.0	341545.0	7621638.6	557.0
LDD025	341471.2	7621706.9	1012.3	341471.2	7621706.9	771.3
LDD026	341544.4	7621708.4	1012.7	341544.4	7621708.4	312.7
LDD027	341609.1	7621708.5	1012.7	341609.1	7621708.5	310.5
LDD028	341609.2	7621628.5	1012.9	341609.2	7621628.5	580.9
PLT008	341500.4	7621675.3	1012.1	341500.4	7621675.3	731.9
PLT009	341513.1	7621738.0	1012.2	341513.1	7621738.0	762.1
PLT016	341610.3	7621630.3	1012.8	341610.3	7621630.3	612.8
PLT017	341545.3	7621640.5	1012.8	341543.6	7621631.8	612.9
PLT018	341610.3	7621790.1	1012.1	341609.5	7621795.0	612.2
PLT019	341681.9	7621710.3	1012.5	341683.3	7621709.7	687.0
PLT020	341609.8	7621710.0	1012.4	341626.7	7621694.7	349.5
PLT021	341545.0	7621710.0	1012.6	341545.5	7621711.8	613.6
PLT022	341470.1	7621710.1	1012.1	341450.0	7621698.5	506.0
PLT023	341550.3	7621780.3	1012.1	341550.3	7621780.3	566.6
REP_001	341110.5	7621702.1	1013.7	341671.1	7621658.3	371.2
REP_002	341579.4	7622199.9	1011.5	341495.0	7621651.0	434.8
REP_004	341063.6	7621743.8	1013.6	341640.8	7621722.2	333.2
REP_005	341628.5	7622167.6	1011.9	341421.5	7621623.1	529.9
REP_006B	341270.0	7622220.6	1012.2	341537.0	7621621.8	375.2

Borehole ID	Top UTM Coordinates			Bottom UTM Coordinates		
	X (mE)	Y (mN)	Elevation (masl)	X (mE)	Y (mN)	Elevation (masl)
REP_007	341939.2	7621890.9	1011.7	341472.9	7621700.3	368.1
REP_008	341235.7	7621748.5	1013.3	341637.1	7621766.6	373.7
REP_009	341073.6	7621739.5	1013.6	341600.3	7621629.6	272.1
REP_011	341230.0	7621750.7	1013.4	341644.6	7621583.6	517.8
REP_012	341941.6	7621880.4	1011.7	341492.6	7621729.0	449.7
Green: Non-interception with UG mine workings.						
Yellow: Will be grouted prior to their interception by mine workings.						

Source: Lucara (2023)

16.4.7.3.2 Predicted Inflow Rates

Figure 16-23 shows the predicted inflow rate to the mine workings from the Base Case Scenario with the following key flow components:

- The total inflow rate reaches a peak of 9,000 m^3/day by the end of 2027 and, thereafter, gradually decreases to 5,700 m^3/day by the end of mining;
- The major inflow components are from the mine development;
- Excluding inflows from dewatering boreholes and ungrouted exploration boreholes, the peak inflow from mine development is 6,500 m^3/day by the end of 2026. The inflow associated with mine development gradually decreases to 3,500 m^3/day by the end of mining;
- The peak total inflow to all identified ungrouted exploration boreholes that could potentially intercept the mine workings is 1,800 m^3/day. The total rate decreases to 600 m^3/day by the end of mining. It should be noted that the peak flow rate of 1,800 m^3/day to the exploration boreholes may vary because of the following factors:
  - The model size of the boreholes is bigger than the actual borehole size, which may lead to larger simulated inflow;
  - The model assumes that there is no resistance of flow along the boreholes, which is not reflective of the actual rough surface condition of the exploration boreholes;
  - Boreholes could collapse; and
  - The flow rate could also be affected by potential geologic structure interception.

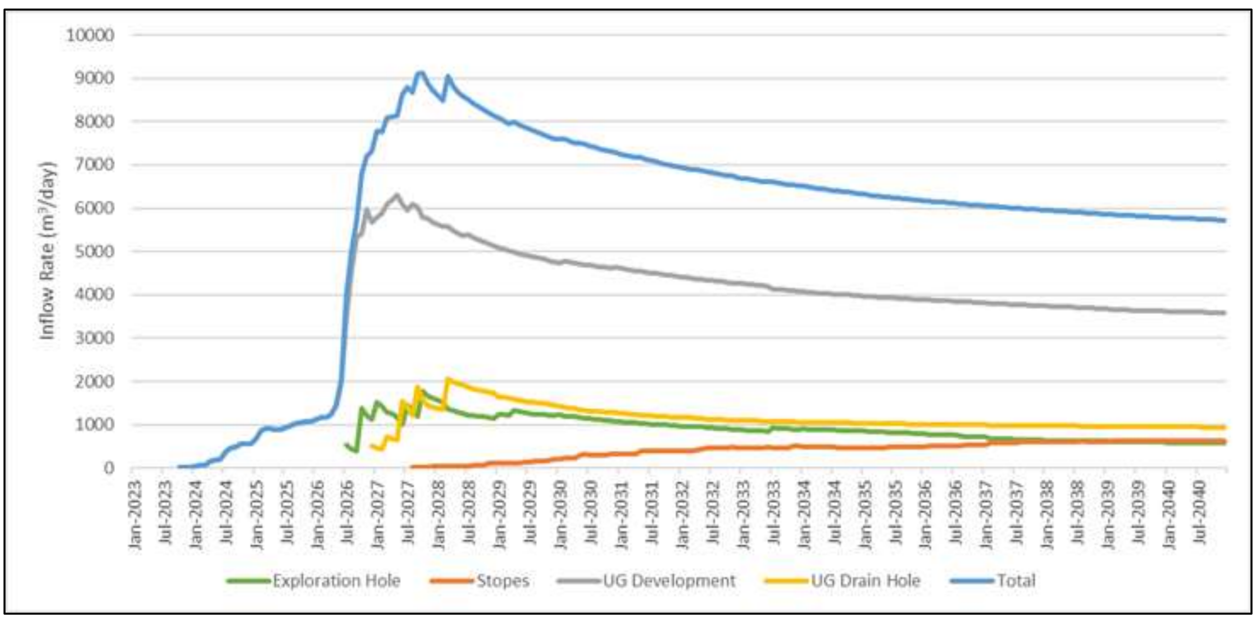
Figure 16-24 shows the predicted inflow rate to different mining levels under the Base Case Scenario. As shown in the figure, the majority of inflow occurs at the 310, 380, and 470 levels as the result of a larger footprint and being in the granite and Mea Formation.

Figure 16-25 shows the predicted inflow rate and key flow components for Scenario 1. Inclusion of 10 additional ungrouted boreholes only increases the peak inflow to the ungrouted boreholes and the total inflow. By the end of mining, the difference of total inflow rates between the Base Case Scenario and Scenario 1 is minor, as shown in Figure 16-26.

Figure 16-26shows the predicted total inflow rates for the Base-Case Scenario, Scenario 1, and Scenario 2. Figure 16-26 shows that, without consideration of future planned dewatering boreholes, the total inflow to the mine workings will increase by approximately 500 m^3/day, which is within 10% of the total predicted inflow by the end of mining. This simulation suggests that the variation of planned dewatering boreholes has a minor effect on the total inflow rate to the UG mining and justifies the use of the planned dewatering rate in the 2021 Model because of the unavailability of the planned future dewatering at the stage of preparation of the feasibility study.

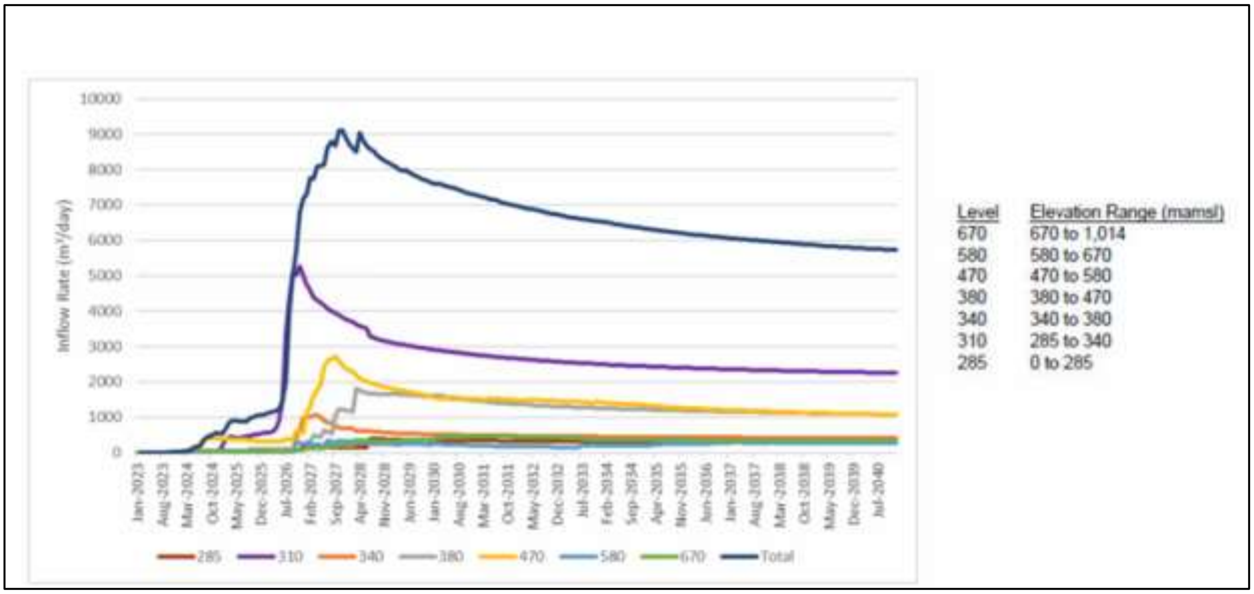
Figure 16-27 shows the predicted dewatering rate for the dewatering boreholes over the LOM. The dewatering rate from the dewatering system decreases from 6,000 m^3/day to 2,500 m^3/day over the UG mining operation for the Base Case Scenario and Scenario 1. Figure 16-27 suggests that the UG mining only slightly reduce dewatering rates of the active dewatering boreholes of the OP. Therefore, it is critical to maintain surface dewatering operations to reduce the seepage to the OP and maintain slope stability during the life of the UG operation.

Figure 16-23: Predicted Total Inflow Rate and Key Flow Components



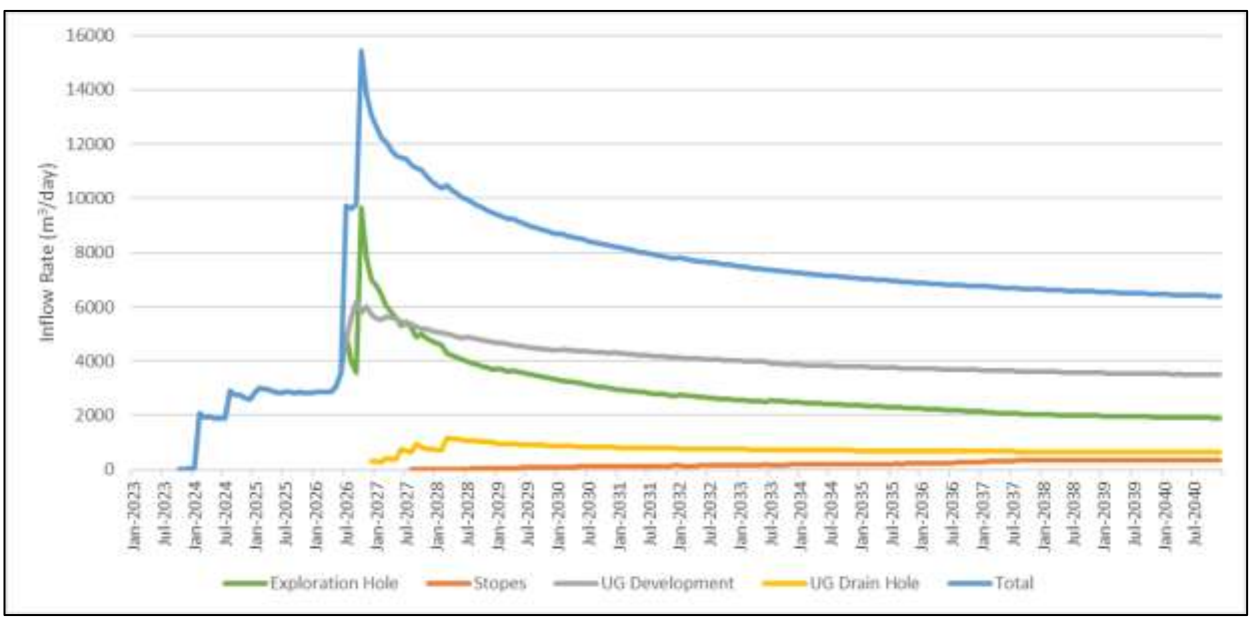
Source: Itasca (2023)

Figure 16-24: Predicted Total Inflow Rate and Flow Rate to Each Mining Level



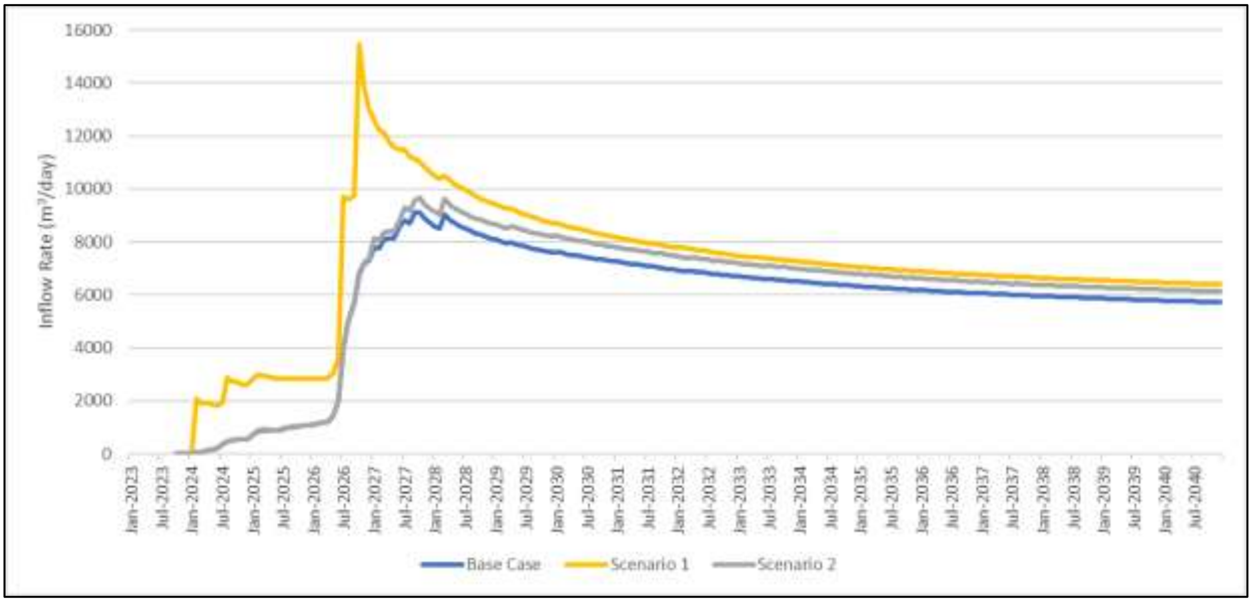
Source: Itasca (2023)

Figure 16-25: Scenario 1 – Predicted Total Inflow and Main Flow Components



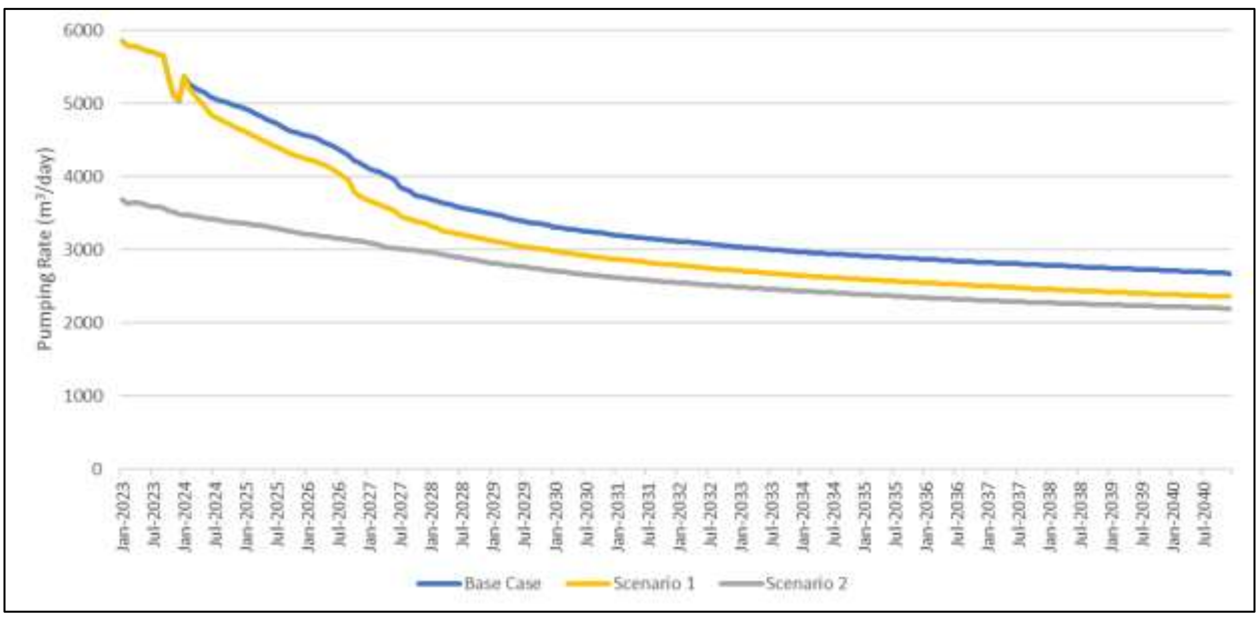
Source: Itasca (2023)

Figure 16-26: Simulated Total Inflow Rate for Base Case and Scenarios 1 and 2



Source: Itasca (2023)

Figure 16-27: Simulated Dewatering Rate from Dewatering Boreholes for Base Case and Scenarios 1 and 2



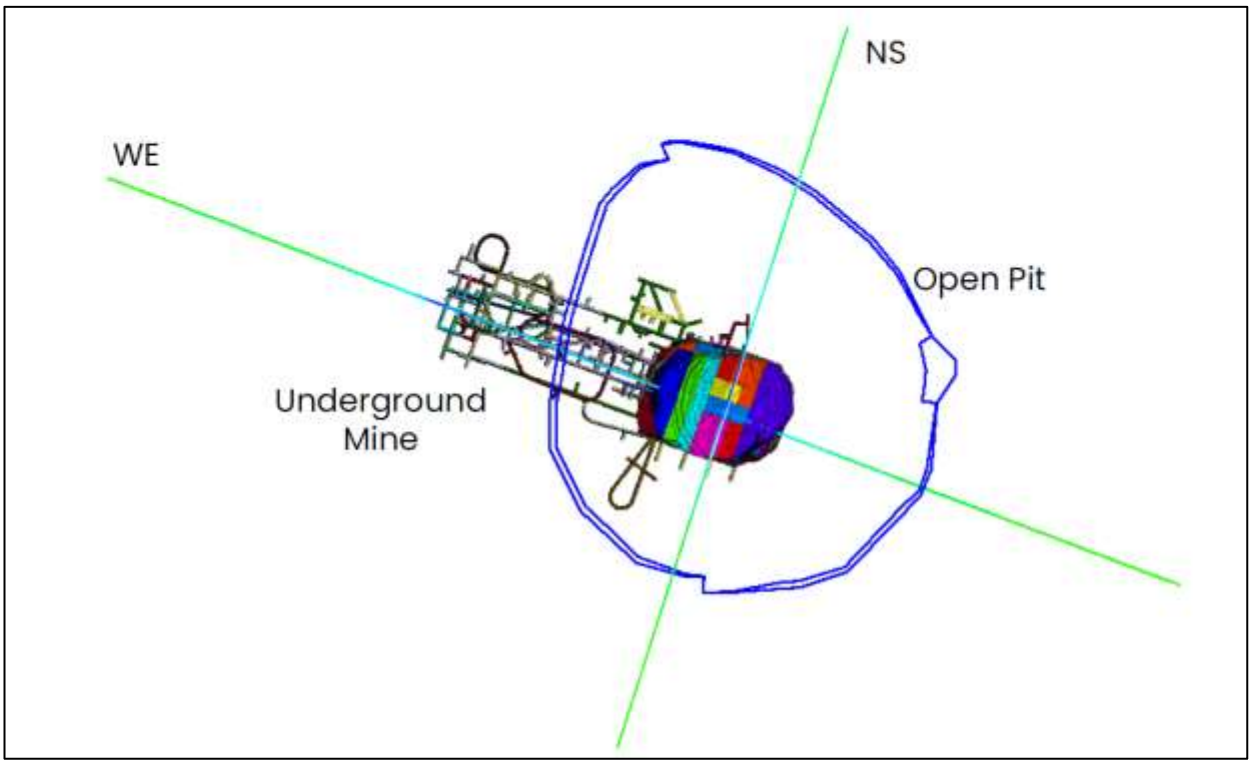
Source: Itasca (2023)

16.4.7.3.3 Predicted Pore Pressure Distribution

The simulated pore pressure distributions for both the OP and UG mining are presented in two cross sections for three mining stages. The locations of these two cross sections are shown in Figure 16-28. Figure 16-29 through Figure 16-34 show the simulated pore pressures in 2023, 2027 (before the stope mining), and 2040 (end of mining) for East-West and North-South sections for the Base Case Scenario. The key observations are summarized below:

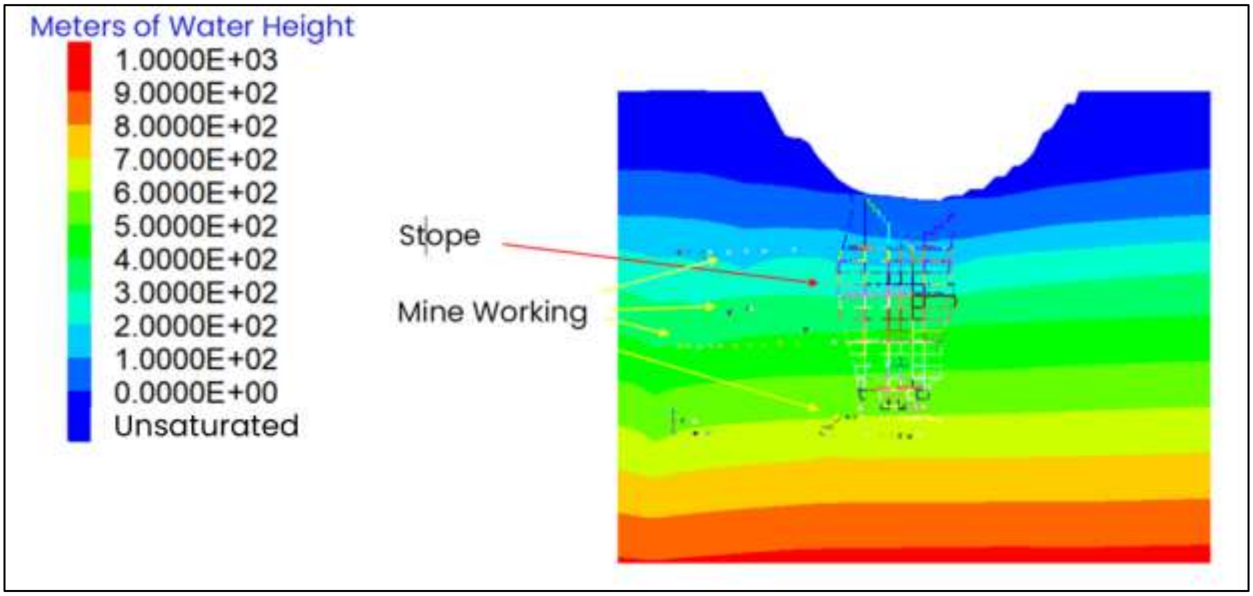
- In September 2023, the pore pressure in the lower portion of the pit is in the range between 0 and 1 mega pascal (MPa);
- Before the start of stope mining in September 2027, the pore pressure decreases noticeably because of the dewatering effect from the development of mine access; and
- By the end of mining, both the OP and UG mines are dewatered and depressurized based on the assumption that groundwater in the UG and dewatering for the OP are actively managed.

Figure 16-28: Locations of NS and WE Cross Sections



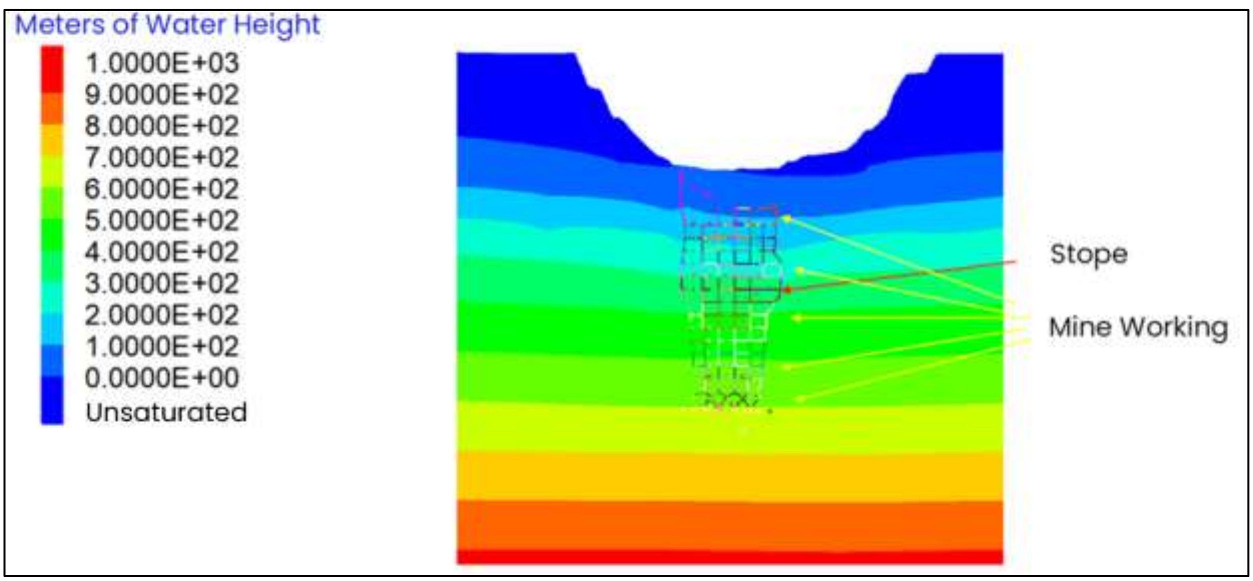
Source: Itasca (2023)

Figure 16-29: Simulated Pore Pressure along East-West Section in September 2023



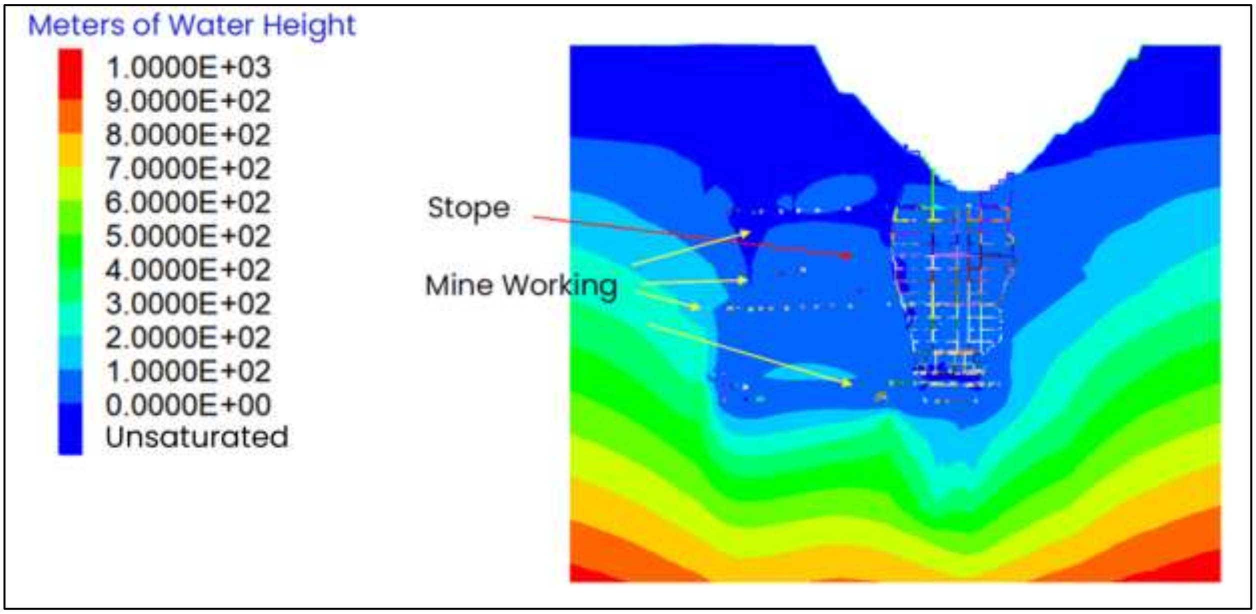
Source: Itasca (2023)

Figure 16-30: Simulated Pore Pressure along North-South Section in September 2023



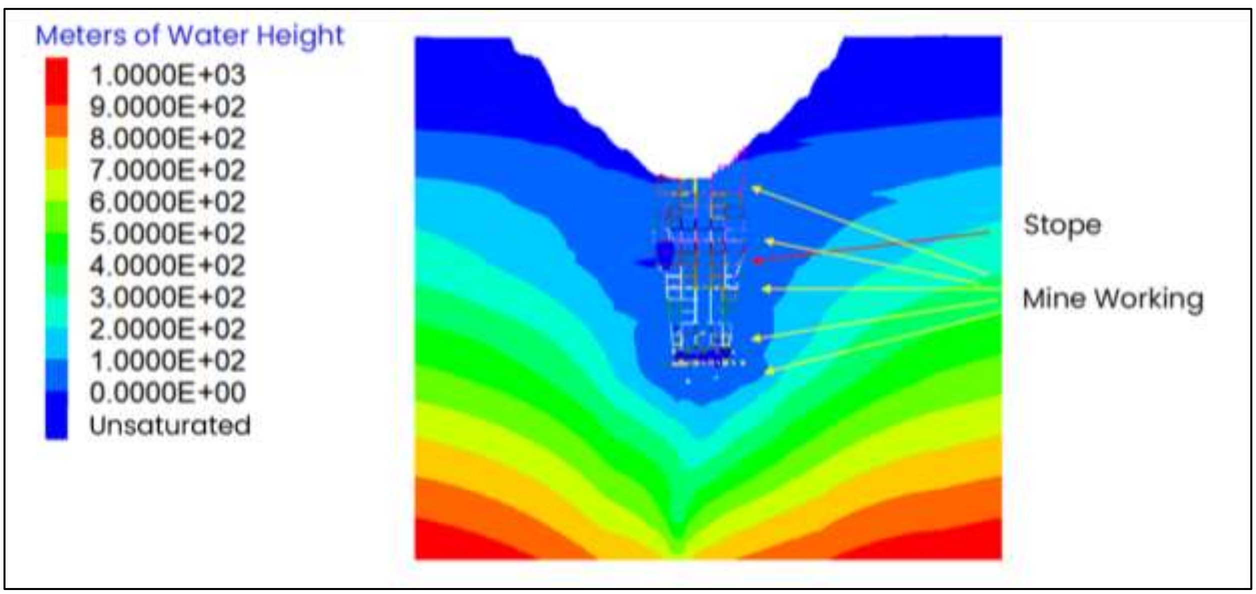
Source: Itasca (2023)

Figure 16-31: Simulated Pore Pressure along East-West Section in June 2027



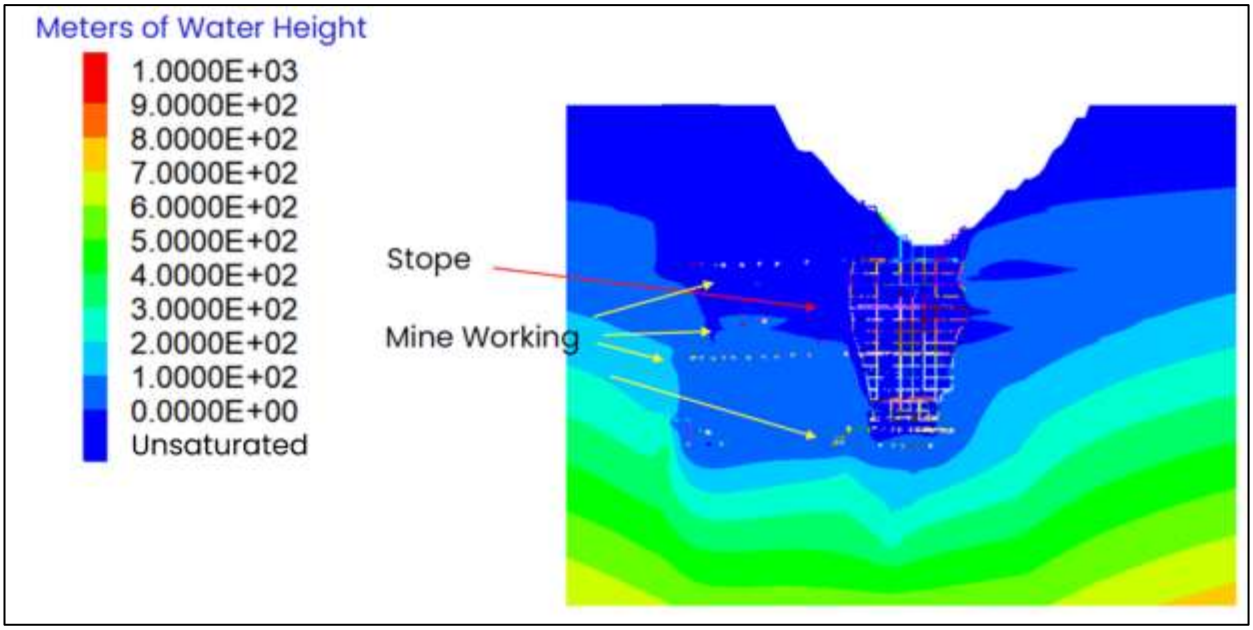
Source: Itasca (2023)

Figure 16-32: Simulated Pore Pressure along North-South Section in June 2027



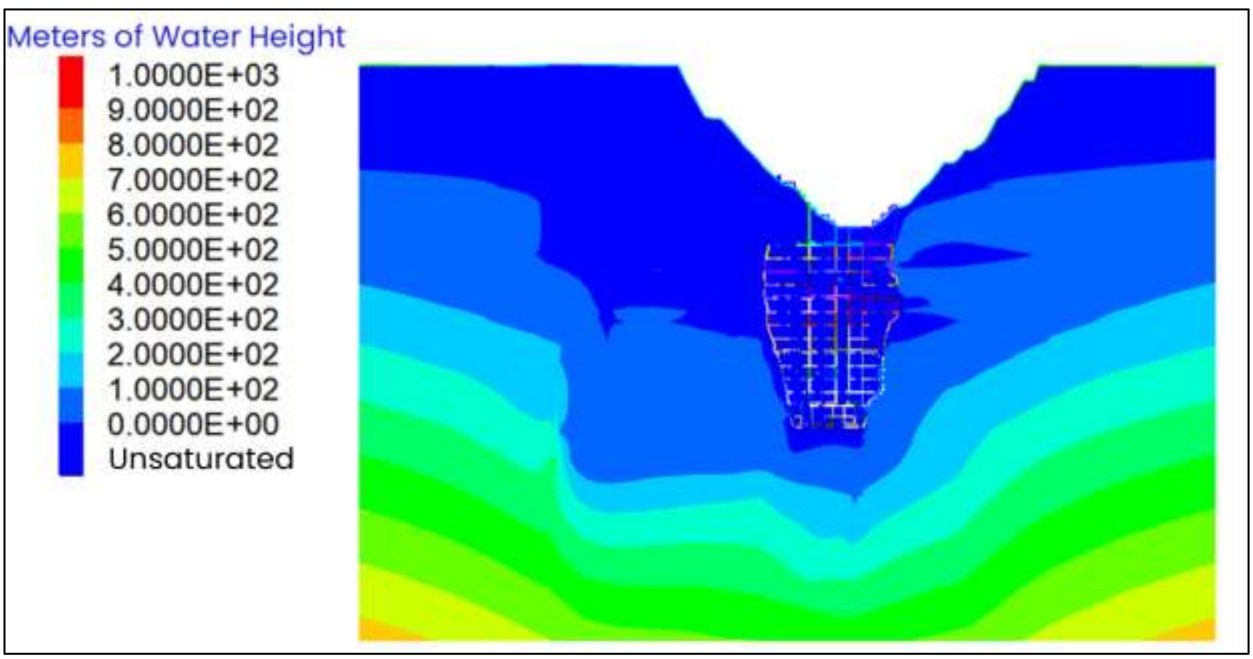
Source: Itasca (2023)

Figure 16-33: Simulated Pore Pressure along East-West Section in December 2040



Source: Itasca (2023)

Figure 16-34: Simulated Pore Pressure along North-South Section in December 2040



Source: Itasca (2023)

16.4.8 Assumptions and Uncertainties

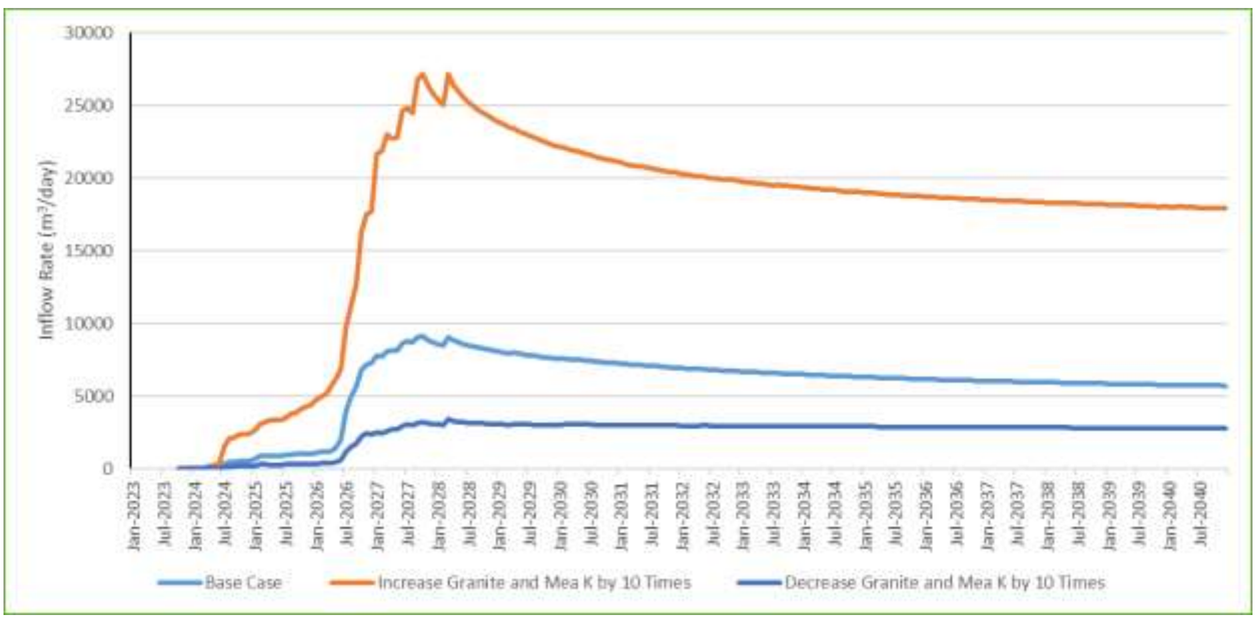
Though the assumptions have been discussed in various sections, the key assumptions and uncertainties related to the predicted inflow are presented in this section because they are critical to risk management.

The surface-water runoff to the OP was not included in the predicted inflow rate to the mine workings. Part of this runoff water will flow to the UG workings. The management of the surface-water runoff is considered in Section 16.8.2.

The measured *K* values in both the Mea and granite units are very limited. Assumptions in the model were made on the distribution of the *K* values along the depth and spatially and the presence of a permeable fracture corridor. Furthermore, there are no monitoring piezometers in the Mea or granite units. The lack of hydrogeologic data poses uncertainty to the groundwater management for the UG mining operation. As shown in Figure 16-35, the simulated inflow rate is sensitive to the *K* values of the Mea and granite units. Increasing the *K* values by 10 times from the Base Case Scenario will increase the total peak inflow rate from 9,000 to 27,000 m^3/day.

In addition to the uncertainty in measured *K* values, there is a lack of measured groundwater levels in both the Mea and granite. The lack of measured groundwater levels also affects the confidence levels of both the model calibration and predictive results.

Figure 16-35: Sensitivity of Inflow Rate to *K* Values in Mea and Granite Units



Source: Itasca (2023)

16.5 Mine Planning Criteria

16.5.1 Basis

Mine Design, Schedules, and Plans shall:

- Incorporate current state of mine construction as at end of June 2023.

16.5.2 Production Rates

Production Rates shall:

- Meet current mill feed rates of 2.7 Mtpa;
- Include design capacity up to 10,000 t/d to account for mechanical and operational availabilities; and
- Be considered to have met the requirements for commercial production after achieving 75% of the daily 7,400 t/d target for a period of 90 days.

16.5.3 Schedules

Mine Schedules shall be based upon:

- 360 days continual operation; and
- 2 x 12 hour shifts per day.

16.5.4 UG Development

All Mine Development shall:

- Maintain minimum 1.0 m on either side of the largest operating mobile machine on the level;
- Incline no greater than 15% on all regularly traveled workings, apart from dedicated conveyor drives which may incline no greater than 17%;
- Incline no less than 1.5% to prevent standing water outside of dedicated sumps;
- Incorporate a 1.0 m radius arched back; and
- Be provisioned with long term ground support in all development drives.

16.5.5 Mobile Mine Equipment

Mobile Mine Equipment shall be:

- Rubber tired; and
- Conventional Tier 3 diesel and electric / hydraulic.

16.5.6 Contractor Support

Contractor Support shall be utilized for:

- Lateral Development;
- Vertical Development;
- Infrastructure Installations; and
- Production Drill and Blast.

Owner's Team shall be utilized for:

- Shaft operation and maintenance (following construction completion); and
- Production Mucking and Comminution.

Table 16-5: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	360
Shifts per Day	Shifts	2
Hours per Shift	Hours	12
Work Roster	On/Off	4/2
Nominal Ore Mining Average Rate	t/d	7,400
Annual Ore Mining Average Rate	Mt	2.7
Ore Density	t/m^3	2.9
Waste Density	t/m^3	Variable by domain (2.9 avg)
Swell Factor	%	36.4

Source: JDS (2023)

16.6 Mining Methods

16.6.1 OP

The currently operating OP at KDM is a conventional load and haul operation. All OP mining operations are performed by mine contractors working year-round on two 12-hour shifts. The on-site mining contractor is currently performing load and haul operations with a Caterpillar 6015 Hydraulic Excavator and Volvo A40 Articulated Dump Truck pairing. The mining contract has a mixed fleet of additional production, support, and ancillary equipment available on-site.

OP mine operations are expected to terminate mid-2025 at an elevation of 713 masl. The mine currently has over three years of stockpiled reserves, which will be consumed as required while the UG mine operations ramp up to commercial production.

The Lucara Botswana mining technical services team has provided the OP production targets, mine plan, and cost inputs used in the FS.

16.6.2 UG

UG mine methods were evaluated in the 2017 Preliminary Economic Assessment completed by Royal Haskoning DHV (RH) (Oberholzer, 2017). This PEA considered block caving (BC), sub level caving (SLC), and longhole open stoping (LHOS) mining methods. SLC with ramp access was recommended due to superior economics, however, geotechnical risks were identified with ramp advancement through stratigraphic units of weaker ground. The PEA identified the need for more detailed trade-off studies to select the appropriate means of UG access and mine method. As a result, in 2018 Lucara Diamonds elected to conduct an internal study to further investigate the mining approach recommended in the PEA, and subsequently commissioned JDS in 2019 to prepare a Feasibility Study (FS) on KDM and re-evaluate the optimal mine method and means of access for the deposit.

The 2019 FS investigated several UG mining methods based on data and information from an exhaustive field program conducted in 2018 and 2019 to define Mineral Resource, geotechnical, and hydrogeological characteristics necessary for making informed decisions at a FS-level study. The small hydraulic radius at depth (27 m), low in-situ (horizontal) stress, and high compressive strength of the kimberlite suggested that the resource will not cave with or without pre-conditioning and will therefore require drill and blast assistance.

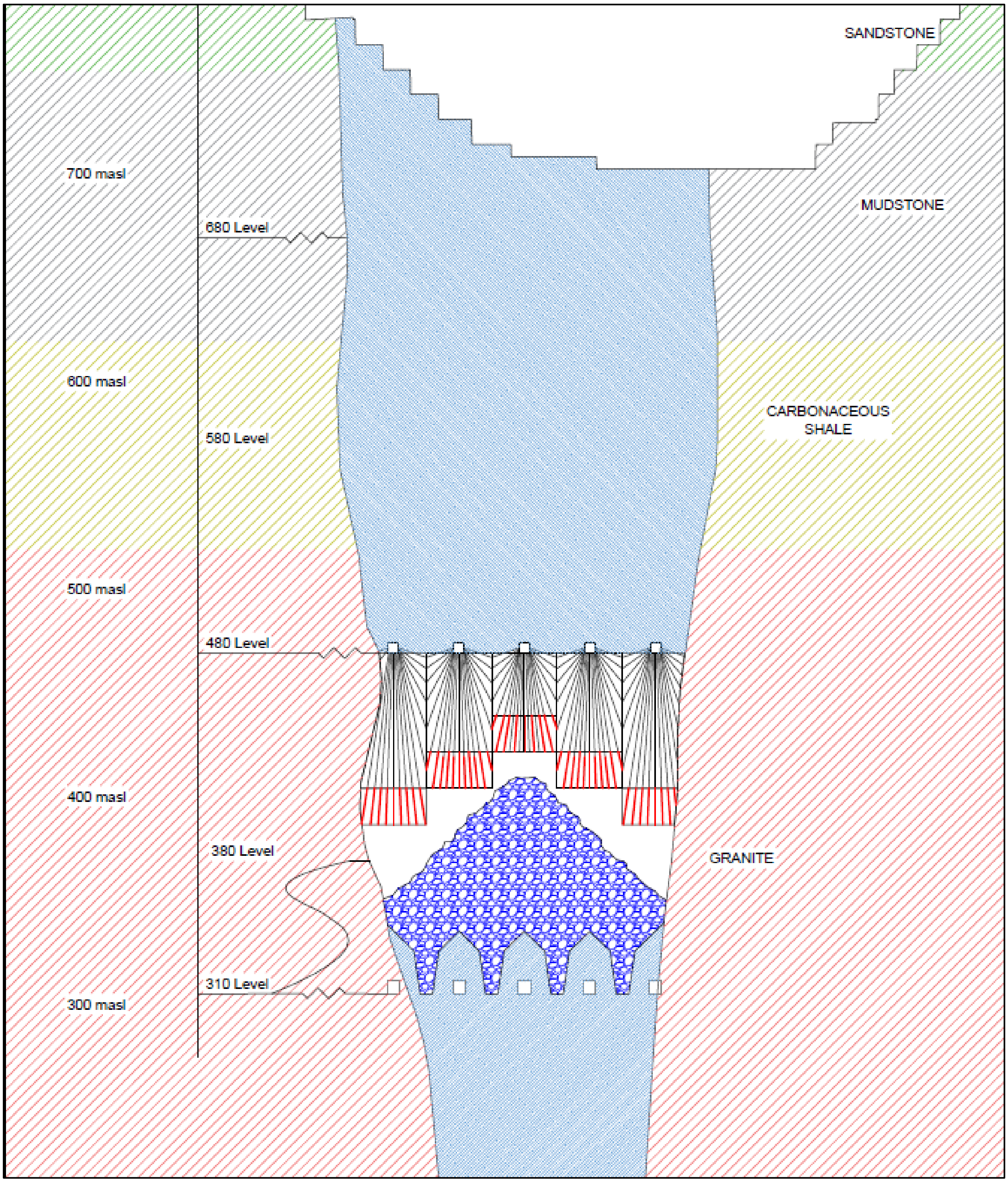
The inability for natural or preconditioned caving to occur has resulted in the development of the LHS mine method, which is essentially a fully assisted cave. The method involves a combination of longhole drilling and blasting to create a large muck pile within the South Lobe, followed by the managed drawdown of the blast material through a panel cave extraction level.

Longhole drill horizons have been designed for the drilling and blasting operations required for this mining method. Drill horizons are spaced at 100 m vertical intervals to accommodate the in-the-hole hammer (ITH) drill's effective drill length of a 150 mm (6") hole.

A pyramidal sequence is proposed for the drilling and blasting of the stopes. This blasting sequence will create a dome shape at the top of the blasted volume to maintain stability of the back. Stopes will be blasted sequentially upwards in 17.5 m increments until a 30 m sill pillar is left between the drill panel and the stope back. A final 30 m blast will wreck this sill pillar and terminate access to the drill panel at that location. The drill will relocate to the next above drill horizon and repeat the process until the lobe is fully blasted. The plan envisions using the same blast hole for multiple blasts, similar to that of a vertical crater retreat mining method.

During drill and blast the broken material will remain within the stope to provide wall support to the South Lobe. The swell created by blasting will be mucked from the drawpoints below the stopes to provide a blasting void, as illustrated in Figure 16-36.

Figure 16-36: Mining Method Illustration



Source: JDS (2019)

Benefits of the LHS mining method include:

- Highest value ore to be extracted first due to the bottom up mining approach;
- Minimal development in weak, water-bearing lithologies near surface;
- Dilution will be delayed (occurring after the payback period) as the weaker host rock is not exposed until later in the mine life;
- Development of the UG mine can occur simultaneously with the OP operations;
- Low operating costs;
- Ease of operation after the drilling and blasting phase is complete and small UG work force requirements;
- Early exclusion of precipitation into the UG workings until the crown pillar is blasted;
- Significant ability to increase production after the drill and blast phase is complete; and
- Designed to manage natural caving should it occur.

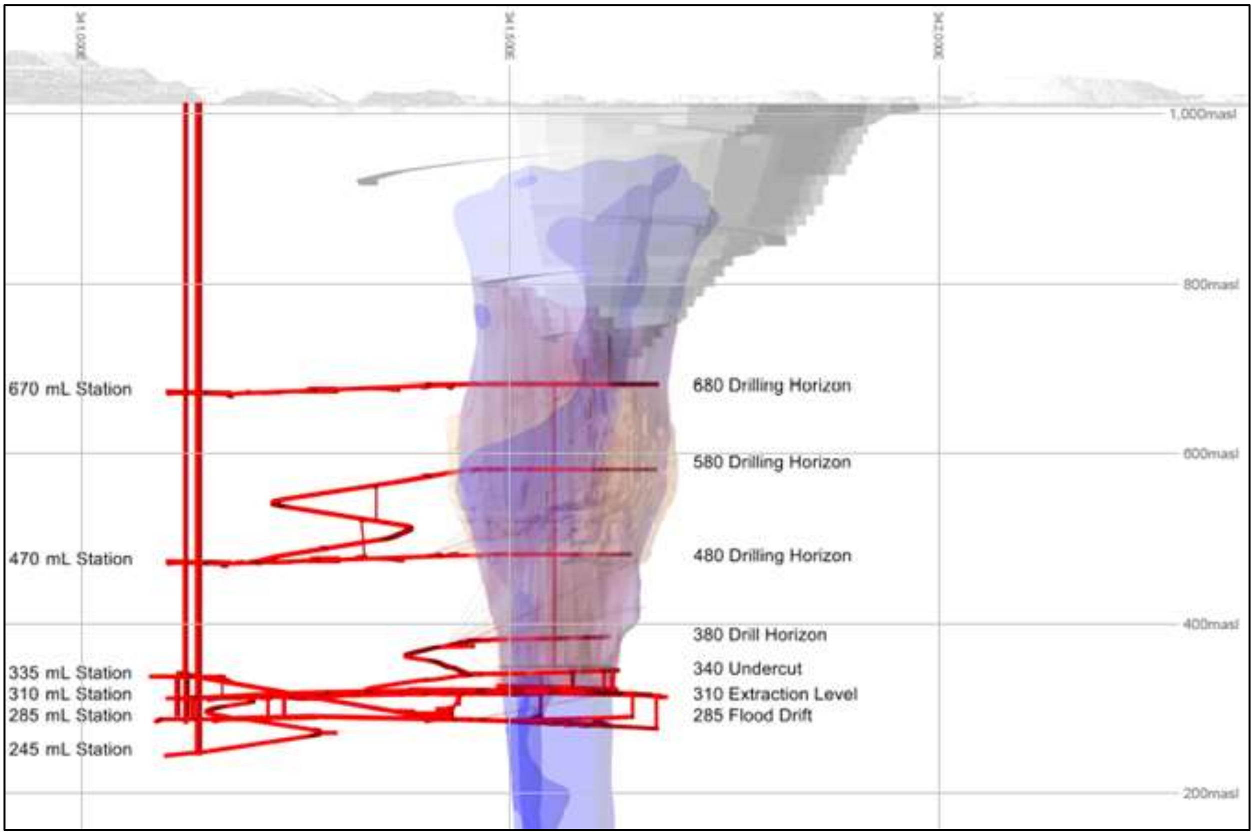
16.7 Mine Design

The KDM UG mine design is based on a panel or block cave layout which supports a bottom-up mining approach and includes the following features:

- Two shafts to provide for all man and material access, ore and waste rock conveyance, and the bulk of mine ventilation;
- A primary Extraction Level at the bottom of the mine workings from which all production ore is mucked;
- Drill horizons from which the ore body is drilled and charged;
- Interconnecting ramps and raises where required to provide man, material, ventilation, and water management connections;
- An UG comminution circuit including crushing, conveying, and skip loading chambers; and
- UG infrastructure required to sustain mining operations including dewatering systems, maintenance facilities, explosives magazines, refuge chambers, and other.

Mine design and scheduling were completed in Deswik software. Figure 16-37 illustrates the LOM development plans for the UGP.

Figure 16-37: Mine Plan Long Section



Source: JDS (2023)

16.7.1 Mine Access

The UG Mine will be serviced by two Shafts located 375 m North-West of the existing OP at a 110-degree bearing, and 100 m from one another. Shafts will be sunk blind using conventional drill and blast equipment and developed concurrently. Shafts will not be frozen or hydrostatically lined and the ingress of groundwater will be managed through in-shaft grouting. Shaft sinking commenced in 2021 and is expected to complete in 2026.

No surface portal or ramp exists or is planned as part of the Mine Plan, as the hydrogeological and geotechnical properties of the near surface lithologies would make such an effort difficult. Shaft access has been selected, in part, for its ability to quickly cross lithological zones of weakness in a relatively quick and controlled manner.

16.7.2 Shaft Design

A single P/S will provide man and material access to the UG Mine. The P/S will be 8.5 m finished diameter, concrete lined, and equipped with rock hoisting skips, man and material cages, and all the UG mine services including power, water, air, and communication lines. The P/S will also serve as the sole fresh air intake to the Mine.

A single V/S of 6.0 m finished diameter, concrete lined, shall serve as the primary return air path and secondary egress for the Mine. No services, conveyances, or ladderways will be equipped in the V/S.

16.7.2.1 Shaft Siting

Shaft locations were selected based on:

- Available geotechnical information and supporting drilling data:
  - Geotechnical holes have been drilled to test, understand, and predict the geotechnical properties of the lithologies to be encountered by the proposed shaft locations, including one dedicated geotechnical drillhole down the centre of each shaft location. See Section 16.3 for details.
- Avoidance of the potential subsidence zone:
  - The geotechnical work carried out, as discussed in Section 16.3, indicates that the inherent stability of the Lobe shape will not cause any significant subsidence. The final excavation shape or subsidence zone of the cave is expected to remain within metres of the actual Lobe shape; and
  - Regardless of the above, a minimum shaft offset for potential subsidence was assumed equal to a 70-degree projection to surface from the extraction level, plus a 100 m buffer.
- Mitigating impacts to the current OP operation:
  - The shaft locations were placed a minimum of 150 m outside of the final pit walls of the OP design.
- Available landscape:
  - The site is already well established with infrastructure including waste dumps, ore stockpiles, processing facility, fine and coarse residue deposition facilities, dewatering wells, camp, and roads. Existing infrastructure was avoided as part of the shaft design criteria.

16.7.2.2 Shaft Headframes

Shaft headframes are of steel construction and built onto a concrete civil foundation. The headframes are not enclosed as there is no need to regulate temperature or air pressure above the collars. The design considers an outer A-frame construction which houses the main sheaves and takes the load of conveyances. Inner tower structures contain the temporary kibble tipping infrastructure used during the shaft sink, which will be replaced with permanent skip and cage conveyance infrastructure. The V/S headframe, upon shaft completion, will be disassembled and removed from site. The P/S headframe will remain for life of mine to service all man, material, and rock movement between surface and UG.

Headframe and collar construction was completed in 2022 as shown in Figure 16-38. P/S and V/S headgears are 61 m and 41 m tall respectively.

Figure 16-38: Production (left) and Ventilation (right) Shaft Headgear

Source: JDS (2023)

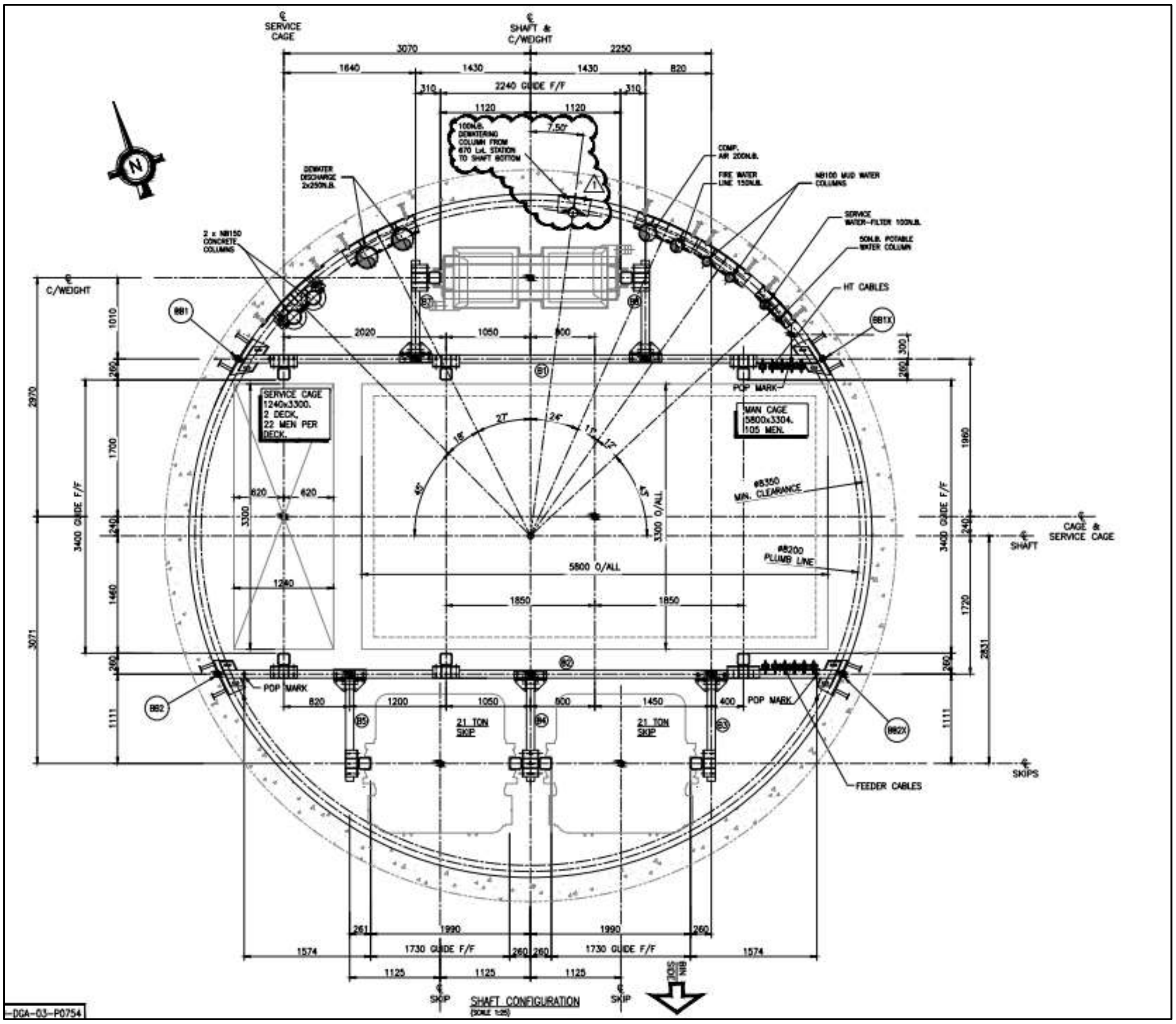
16.7.2.3 Shaft Capacity

The P/S shaft has the following design elements and capacities:

- Two (2) skips with payload capacity of 21 t each, capable of hoisting 3.2 to 3.5 Mt/a;
- One (1) Service cage of internal dimension 5.5 m x 3.0 m and payload capacity of 105 men or 17.5 t in-cage, and 27 t with a heavy lift bridal; and
- One (1) auxiliary cage with 16-man capacity.

The P/S skips, cages, and counterweight will all operating on fixed steel guides. Shaft internal arrangement can be seen in Figure 16-39 and will be such that the two cages reside back-to-back with the skips on the south wall and counterweight on the North. Shaft services will reside primarily on the North wall adjacent to the counterweight.

Figure 16-39: Production Shaft Internal Arrangement



Source: LUCKAR05E-1261-S-DGA-P0754, Permanent Shaft Configuration, UMS (2023)

16.7.2.4 Shaft Hoists

Shaft hoists are electrically driven and housed indoors for protection from the elements. Hoists are designed to meet the lifting requirements of each shaft conveyance and outlined in Table 16-6. Hoists are operated on three eight-hour shifts daily in accordance with local regulations. Hoist controls are unique to each winder and are located within the winder buildings in separate temperature-controlled rooms.

Table 16-6: Hoist Design Criteria

Hoist	Hoist Details				Suspended Masses (kg)			Rope Data	
	No. Drums	Power (RMS) KW	Drum Dia (m)	Rope Speed m/s	Conveyance and Attachments	Payload (kg)	Total End Load (kg)	Dia (mm)	Length (m)
Sinking									
Production Shaft									
Kibble	2	4,800	5.59	12.53	6,860	14,000	20,860	46	1,725
Stage	2	380	2.5	0.23	86,000	51,650	137,650	41	3,400
Ventilation Shaft									
Kibble	2	1,492	3.07	6.89	5,272	8,000	18,396	40	1,470
Stage	2	190	3.48	0.25	67,200	51,567	118,767	41	3,265
Permanent									
Production Shaft									
Skip	2	4,800	5.59	13.71	10,500	21,000	31,500	56	1,330
Man and Material Cage	2	2,400	5.59	8.7	12,600	17,500	30,100	61	1,185
Aux Cage	1	700	2	6.75	4,600	1,500	6,100	26	1,003
Ventilation Shaft									
N/A									

Source: LUCKAR05E-TDR-UMS-0002, Winder Duty Summary Sheet, UMS (2022)

16.7.2.5 Shaft Stations

The shafts will service seven (7) shaft stations, as listed in Table 16-7 below. Several shaft stations will be connected to one another through ramps or connections. Some shaft stations are captive such that the only means of access to that level is through the shaft.

V/S stations will not be equipped with permanent services apart from ventilation bulkheads, regulators, protective screens and gates, and a drawbridge which may be used to load passengers into an egress capsule in the event of emergency.

P/S Stations will be equipped with permanent services including air, power, and water. Incoming water lines will be provisioned with pressure reducing stations r. Station protective steelwork will be used as the mounting points for thrust blocks on incoming and outgoing water lines.

Station floors will be concreted and reinforced with rail mats to protect the curb. Rail and drop posts will be imbedded in the floor to allow for rail car loading and unloading of materials. Farm gates designed to withstand the force of an LHD will be erected at the entry of each Shaft Station. D-plates will be mounted to the walls of the station to provide lifting points for large loads and mobile equipment.

Water rings will be installed above the brow of each P/S station to collect groundwater leaks in the shaft and direct this water to a sump located on the station. V/S water rings will not be serviceable in permanent condition given a lack of permanent shaft conveyances, however, water rings will be fabricated and installed as needed above Vent Shaft stations during the shaft sink to mitigate nuisance water.

Table 16-7: Shaft Stations

Station	V/S	P/S	Servicing	Host Rock
718	x		718 L Slinging Cubby	Mudstone
670	x	x	680 L Drill Horizon	Mudstone
470	x	x	580 L Drill Horizon 480 L Drill Horizon	Granite
335	x		335 L Fine Ore Storage	Granite
310		x	380 L Drill Horizon 340 L Undercut 310 L Drawpoints	Granite
285	x	x	285 L Skip Loading 285 L Flood Drift 285 L UG Crusher Access 245 L Shaft Bottom	Granite
245		x	245 L Shaft Bottom	Granite

Source: JDS (2023)

16.7.2.6 Skip Loading

The skip loading station will be conducted at the 285 L station.

Two fine ore bins of each 7.5 m diameter, 42 m height, and 2,400 t capacity will contain rock processed through the UG crusher and development muck passes. The fine ore bins will be collared at the 335 L Station and terminate at the 285 L Station, and feed directly into the shaft skip loading equipment. At full production (2.7 Mtpa) the fine ore bins will provide for 18 hours of storage capacity. Bin Capacity calculations are shown in Table 16-8 below.

Table 16-8: Fine Ore Bin Sizing

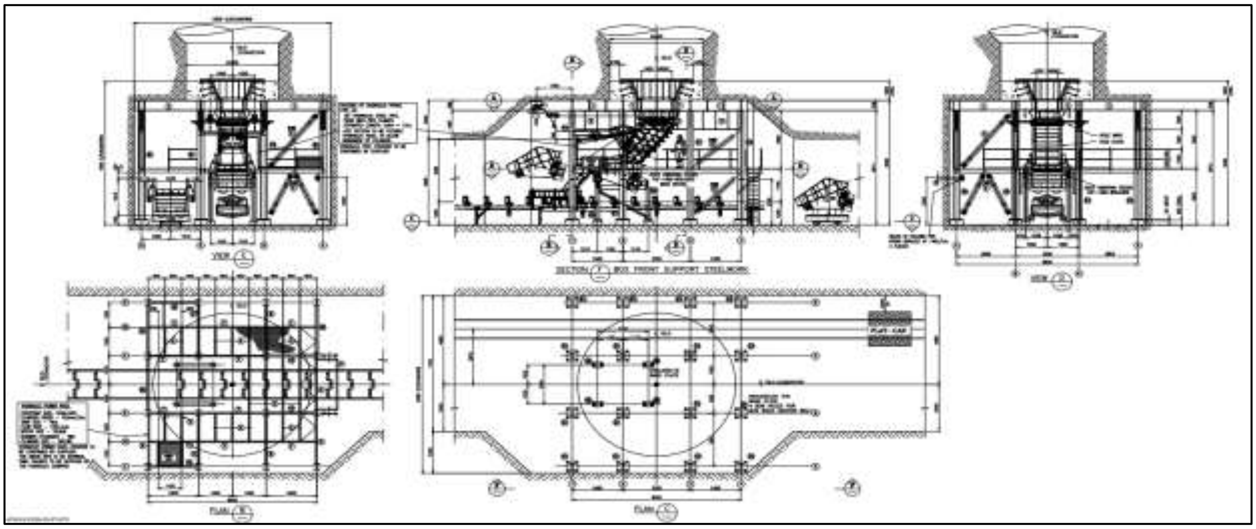
Item	Units	Value
Collar Elevation	masl	335
Destination	masl	285
Load Out Drift Height	m	8.5
Silo Length	m	42
Silo Diameter	m	7.5
Silo Volume	m^3	1,855
Fill factor	%	75
Broken Density	t/m^3	2.0
Silo Capacity	t	2,783
# of Silos	#	2
Total Capacity	t	5,566
Mine Throughput	t/d	7,397
Storage Capacity	hours	18

Source: JDS (2021)

The Fine ore bins shall be constructed using a raise bore slot cut followed by slipping via single deck stage, winches, and pneumatic hand drills. The raise bore slot will serve as pilot, ventilation, muck pass, and dewatering.

At the discharge, the bins will be fitted with a concrete bulkhead supported by steelwork designed to take the weight of the bin contents, as well as provide tie points to the chutes, arc gates, vibrating feeders, power packs, and maintenance hoists, as shown in Figure 16-40.

Figure 16-40: Fine Ore Bin Bulkhead



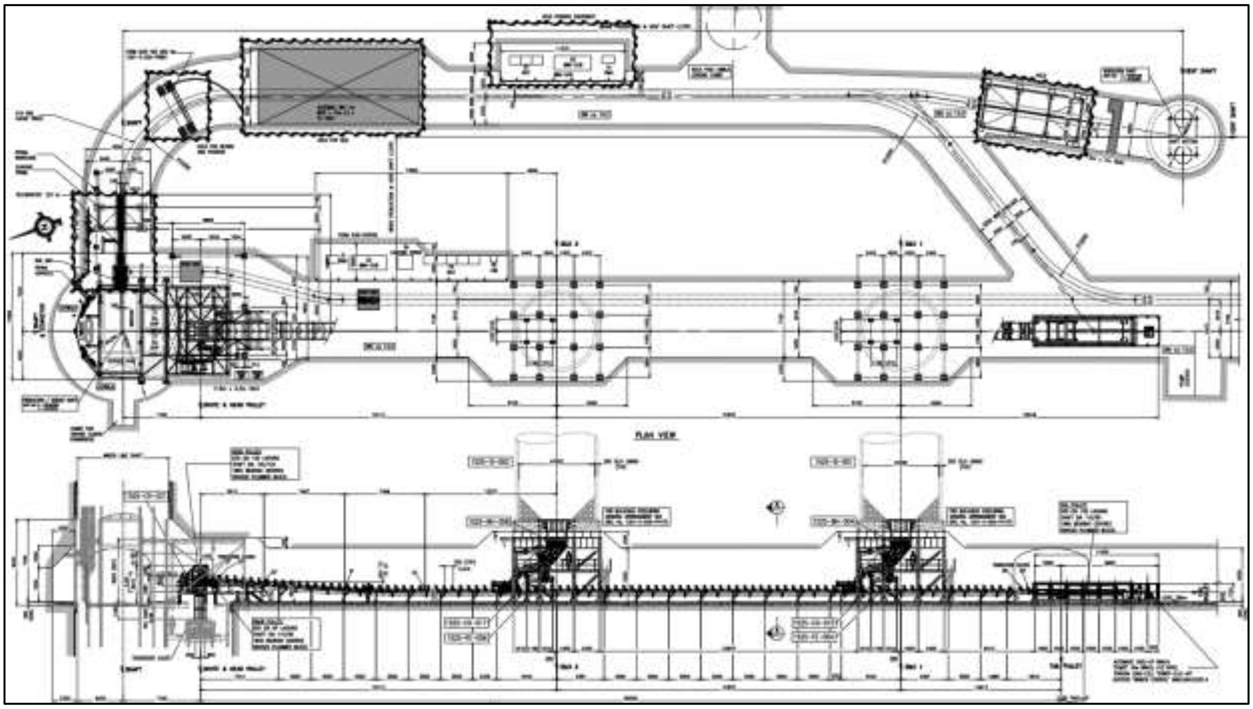
Source: LUCKAR05E-1261-S-DGA-P1470, Bulkhead Steelwork General Arrangement (UMS 2022)

The skip loading conveyor will report material to a traversing chute and into one of two loading flasks. Flasks will be of steel construction and located within a 13 m tall loading pocket excavated into the side of the shaft.

Loading flasks will deposit rock into one of two 21 t skips. Skips will be 12 m tall, of aluminum construction, and come equipped with top and bottom guide rollers. Hook attachments at the skip bottom will allow for suspending an inspection basket.

The skip loading station will be furnished with parallel drives. One drive will contain the skip loading conveyor terminating at the loading flask, and the other will serve as man and material bypass terminating at the main cage entry. Embedded rail alongside the conveyor will provide access for rail car to maneuver liner plates, conveyor belt, and other wear materials to service the skip loading equipment.

Figure 16-41: Skip Loading Station Feed Conveyor Layout



Source: LUCKAR05E-1261-M-DAL-P0861, Skip Loading Station Feed Conveyor Layout, UMS 2023

16.7.2.7 Shaft Bottom

V/S Bottom will be situated at the 285 L station. This station will serve as the primary return air route for the mine in permanent configuration. The shaft bottom will be equipped with a barricade to prevent unauthorized entry and concrete floor which is sloped to direct shaft leakage to a nearby sump.

P/S Bottom will be situated at the 245 L station, 40 m below skip loading. A cat ladder will be installed within the P/S to provide access from the skip loading station to the loading pocket, and onto shaft bottom for maintenance purposes. Cage guides will not extend to shaft bottom, and as such access will be limited to ladderway or ramp from the 285 L. A small sump will be located immediately off-shaft to collect and direct water to the sump on 285 L. All pumps, fans, and lighting will be powered from a substation located on the 285 L to prevent electrical damage in the event of shaft flooding.

Skip hoisting systems are not 100% efficient and are known to spill small amounts of muck into the shaft cavity during the skip loading operation. Spillage may occur as the result of several factors including:

- Skip Loading Commissioning;

- Overfilling of the skips;
- Poor alignment of chute and skip; and
- Skip Leakage.

It is anticipated that upwards of 0.1% of all material hoisted will report to the shaft bottom as spillage, which at full production yields 2,700 t annually, or more than 20 m of equivalent shaft depth.

Left unmanaged, shaft spillage can become a hazard that impacts not only production but safety.

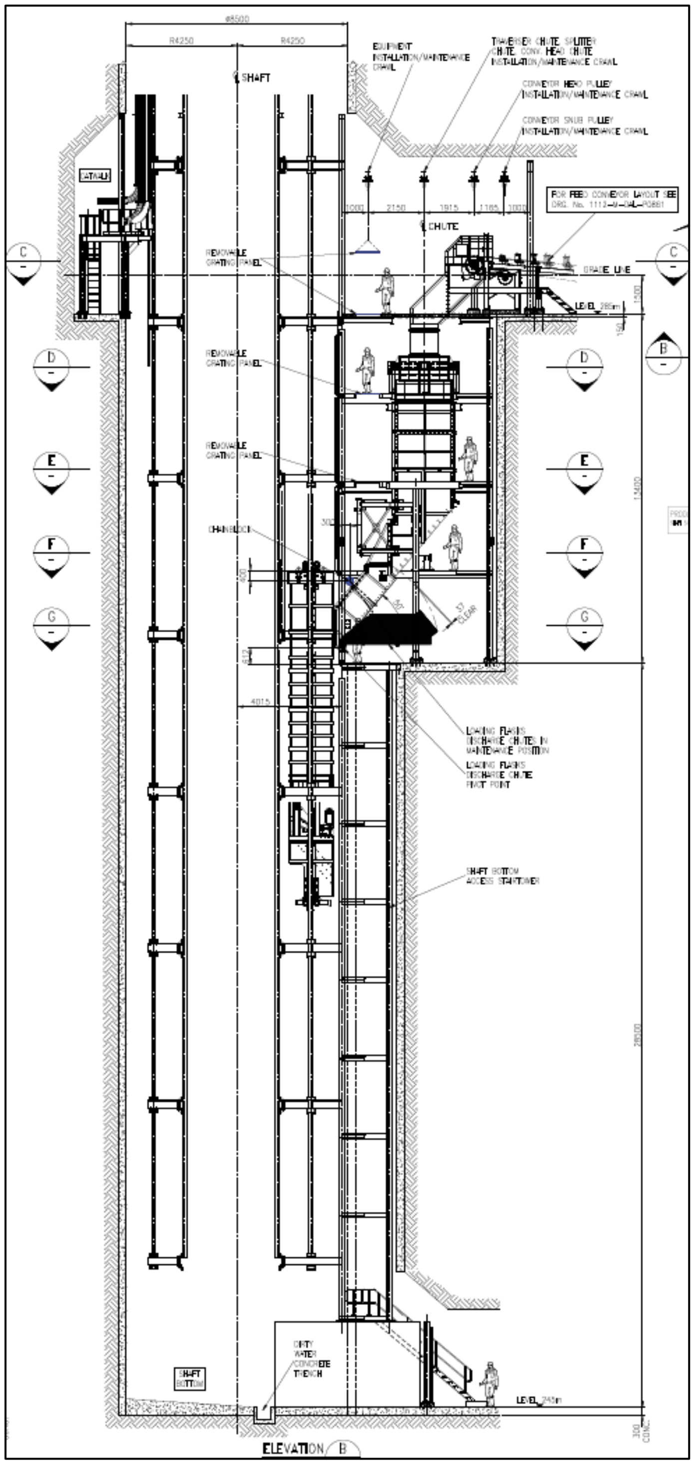
Under current design should the shaft be filled with more than 15 m of muck the skips will no longer be able to seat under the loading flask. Spillage should not be allowed to accumulate to this degree, however, as it would bury shaft bottom pumps and cause shaft flooding.

It is therefore critical that the P/S be equipped with suitable spillage handling systems with commissioning, and certainly prior to full production. Spillage handling systems typically comprise of either:

- Ramp access to shaft bottom;
- Shaft spill pocket; or
- Shaft spillage hoist.

A ramp to P/S shaft bottom will serve as maintenance access to shaft bottom pumps, as well as LHD access to muck out shaft spillage. Spillage will be trammed up the shaft bottom ramp and deposited directly onto the skip loading conveyor via a tail pully loading system which will be employed during initial lateral development. Alternately spillage may be trammed to the crusher tip, once commissioned, and report to the fine ore bins with crushed rock. Until the shaft bottom ramp is driven, the cat ladder will be used to access shaft floor and a miniature remote excavator (Brokk) shall muck shaft spillage into buckets that are slung underneath the skip inspection basket.

Figure 16-42: Skip Loading Pocket



Source: LUCKAR05E-1112-M-DAL-P1423, Skip Loading Station Equipment Access and Maintenance Layout, UMS (2021)

16.7.3 Stope Design

The South Lobe is over 700 m in height and at the narrowest point is 100 m in diameter. The ore zone is continuous, entirely economic, and lends itself to bulk mining. The stopes are therefore not limited as much by geometry or physical boundaries as they are by equipment capabilities and geotechnical requirements. Stopes have therefore been designed to maximize the effective length of long hole drilling equipment and minimize capital development requirements of sub levels.

16.7.3.1 Sublevel Spacing

The effective downhole reach of a Sandvik DU311TK In the Hole Hammer (ITH) drill equipped with 150 mm (6”) bit is greater than 100 m and has been used to establish a 100 m sub level spacing.

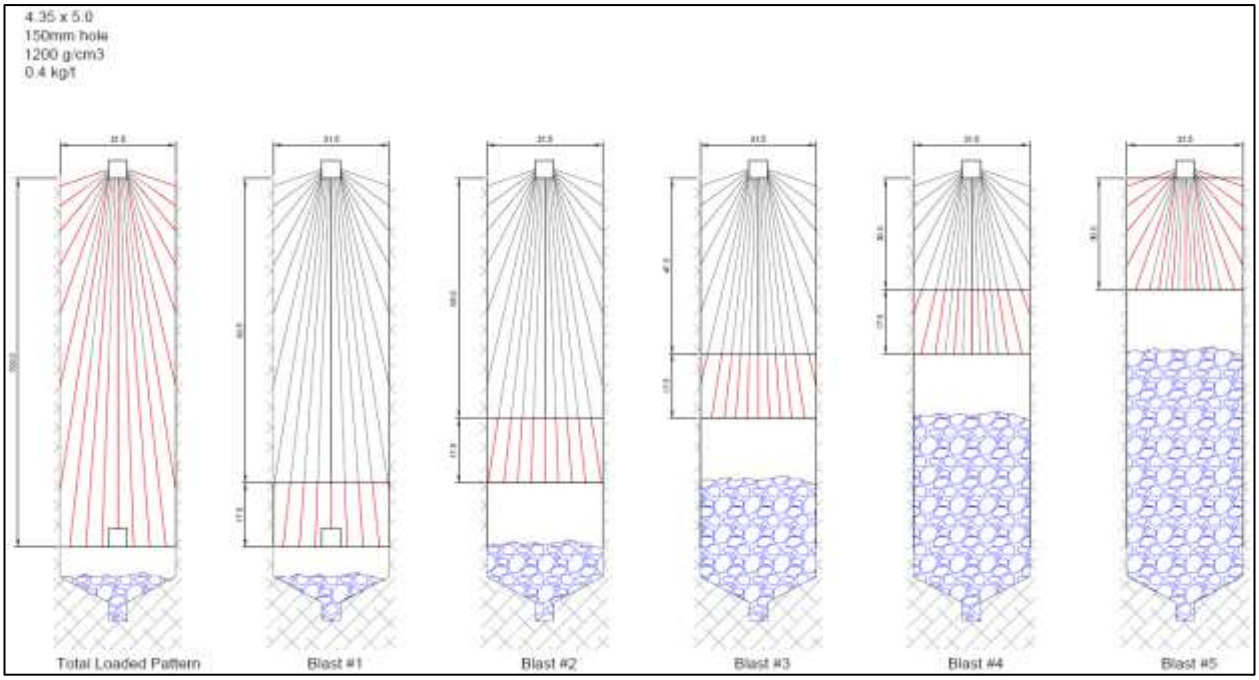
16.7.3.2 Drill Pattern

The KDM OP utilizes a 0.3 - 0.4 kg/t powder factor and achieves excellent fragmentation with more than 90% passing 400 mm (Fragmentation Report Summary from May to October 2021, Lucara 2021). UG stope drilling will be designed to achieve a similar powder factor with the use of 150 mm drillholes and a burden and spacing of 4.35 m and 5.00 m respectively. With these parameters the average length of hole per 100 m tall stope will be 58 m, with an average 34 t/m drilled.

Below the first drill horizon (380 masl) a powder factor of 0.6 kg/t will be used to ensure high rock fragmentation at the start of the shrinkage process. This will be achieved by using the same burden and spacing but with a 165 mm (6.5”) drill bit instead of 150 mm as used on levels above.

100 m tall stopes will be drilled in a downwards fan pattern. Stopes will be blasted in increments until a 30 m sill pillar remains and is ultimately wrecked during level abandonment. Figure 16-43 illustrates a typical blast pattern.

Figure 16-43: Production Blast Pattern



Source: JDS (2019)

16.7.3.3 Stope Modelling

Stopes have been designed using 3D Mine Planning software Deswik. Stopes designed to date have been done so by slicing and appending a diluted resource wireframe.

16.7.3.4 Stope Dilution

Planned and unplanned dilution has been accounted for by including dilution halos around the South Lobe resource Wireframe prior to stope modelling.

Unplanned dilution, resulting from poor drill practice or unplanned geotechnical conditions has been accounted for with a 1.0 m dilution halo.

Planned dilution, resulting from unstable ground conditions was predicted through FLAC3D geomechanical modelling by Itasca (Itasca 2021). This modelling suggests 2.7 Mt of host rock (7%) will report to the open stope as mining progresses through the Tlapana carbonaceous mudstone complex and has been accounted for in stope designs within this unit.

Internal stope dilution is limited to waste blocks or inferred resource contained within the stope designs. Within the South Lobe exists an inferred kimberlite domain, KIMB3, which has been treated as zero grade waste blocks within stope design.

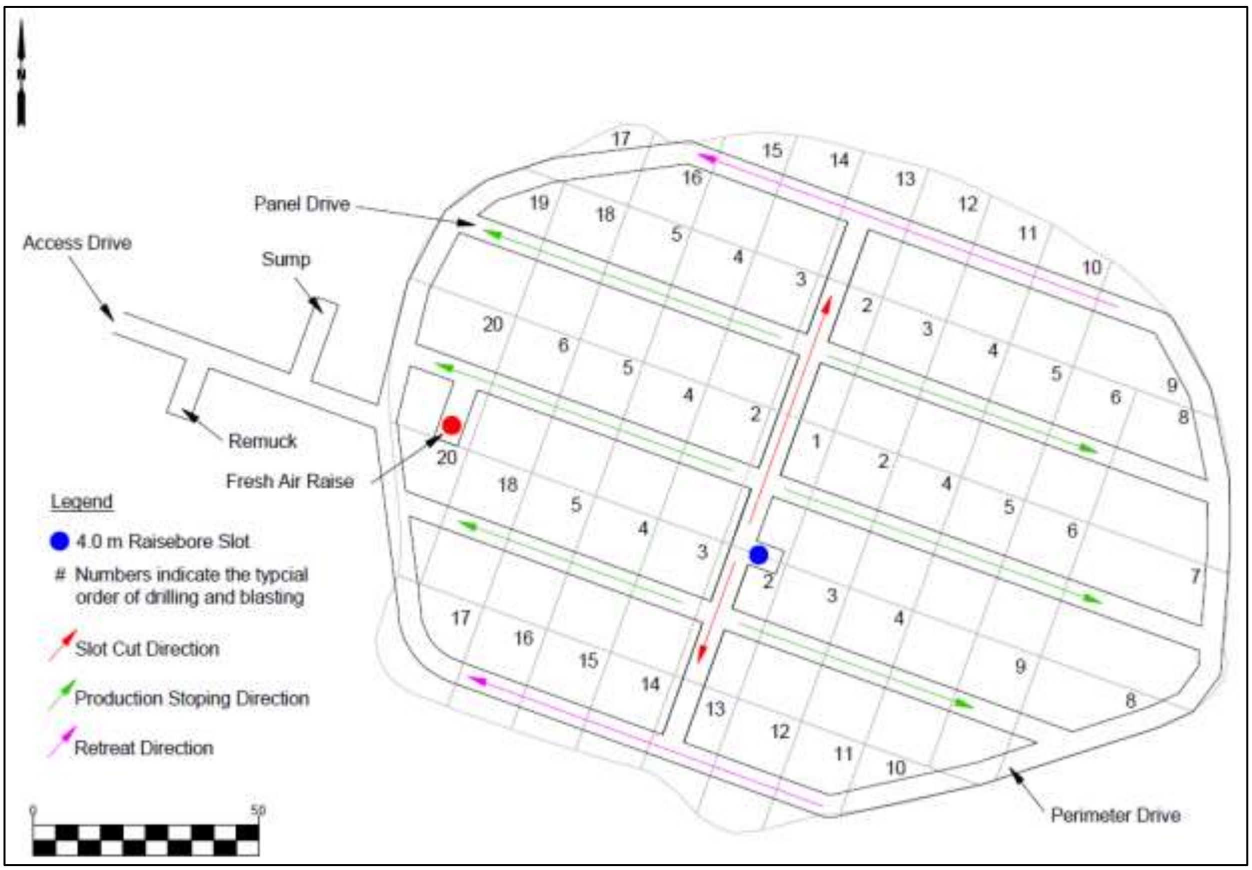
16.7.3.5 Stope Recovery

Stope designs include a 100% Mine Recovery.

16.7.3.6 Stope Sequencing

A slot raise will provide the initial blast void and free face for the long hole stopes to break into. A crosscut will be developed across the centre of the lobe, perpendicular to the direction of the drill panels on each drill horizon. A slot raise will be driven vertically between these crosscuts and will be systematically slashed out using a long hole drill to provide a slot cut across the lobe. The slot will be stopped short of the perimeter drive on each horizon to provide man and equipment access to the back side of the drill panels. Long hole stopes will then be drilled and blasted in retreat from the centre of the lobe, following a pyramidal blast sequence. Figure 16-44 illustrates in plan view the stoping sequence on a typical drill horizon. Figure 16-36 illustrates a cross section of the south lobe, showing the pyramidal advance of stopes. In this figure the central stope is loading the final blast to wreck the sill pillar at that location.

Figure 16-44: Plan View of Typical Blasting Sequence



Source: JDS (2019)

16.7.3.7 Design Optimization

Stopes have been largely designed around geotechnical constraints and the need to maintain a dome shape in the back while blasting. Should geotechnical conditions permit larger brows, or steps, between blasts there may be opportunity to increase stope dimensions in the X, Y, and Z direction to improve drill and blast efficiencies. The stope drilling and blasting design is very flexible and lends itself to optimization as the operation ramps up.

16.7.4 Extraction Design

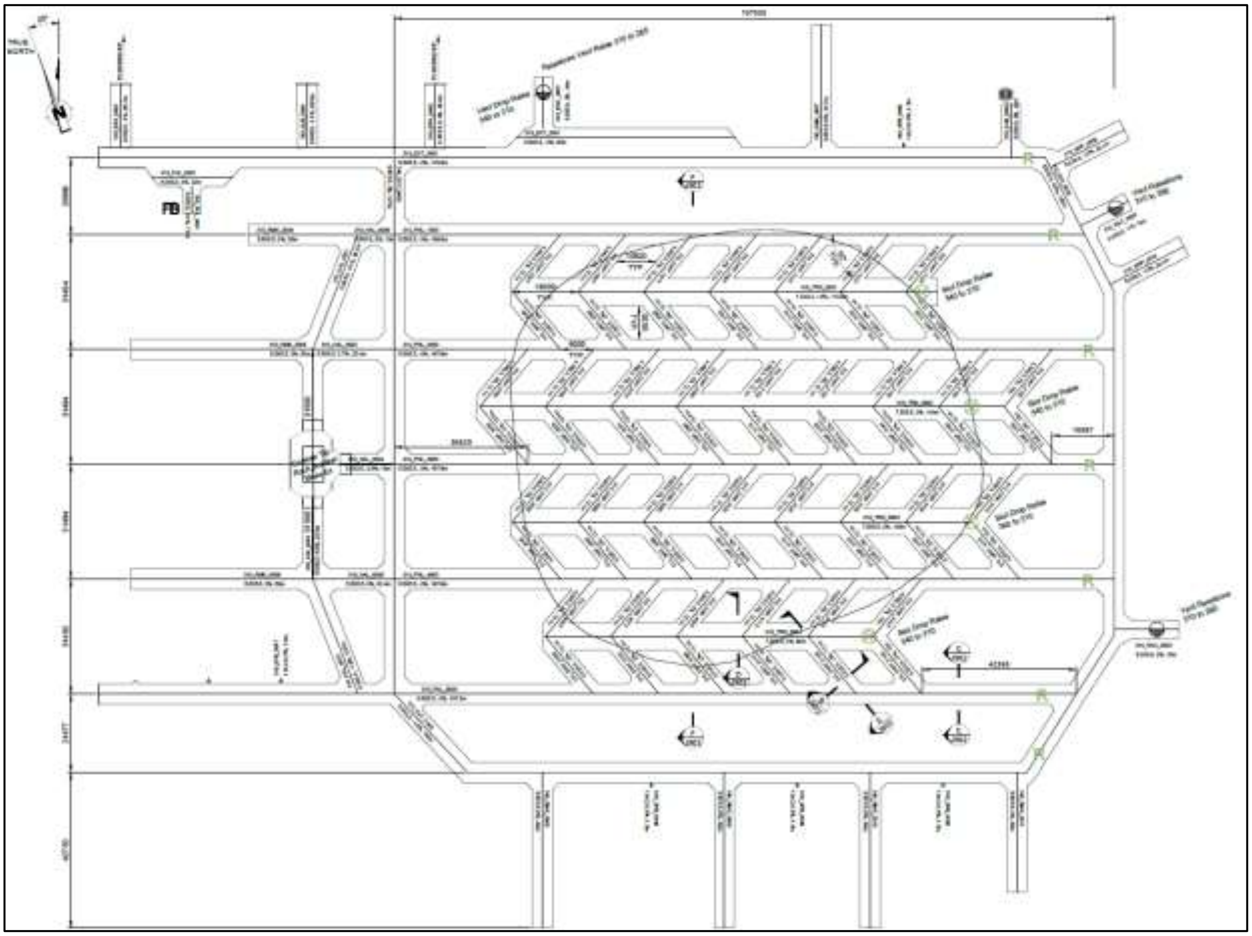
The extraction level will contain the drawpoints from which all production ore is extracted from the stopes. The level is designed with the following features as seen in Figure 16-45:

- 1x Perimeter Drive offset from the South Lobe by 15 m;
- 5x parallel Extraction Drives crosscutting the South Lobe from West to East, spaced 31.5 m apart;
- 4x continuous Troughs and Major Apex Pillars between the Extraction Drives;
- 50x drawpoints between the Extraction Drives and Troughs driven on 28 m spacing in an offset herringbone pattern;
- 1x double wide central Grizzly Tip with 3-way access from the Extraction Drives; and
- 7x Remuck Bays around the Perimeter Drive with capacity for two day's production

The proposed design allows for maximum draw control of the blasted ore, whereby operations will utilize numerous drawpoints to manage the shape of the muck pile and reduce preferential draw of dilution. The design allows for continuous mucking to keep the muck pile in motion at all times, minimizing risk of re-compaction or creating a deadweight above the extraction panels. Constantly drawing from each drawpoint minimizes the risk of a mud rush or water rush by mixing any pockets of water that may have developed within the muck pile with dryer material.

Storage capacity has been designed into the mine plan to allow for constant movement of material from the drawpoints in the event of a material handling shutdown (planned or unplanned). 280 m of dedicated remuck capacity located immediately adjacent to the extraction drive will permit 1 bucket of material drawn from each open drawpoint per 24-hour period for up to 10 days.

Figure 16-45: Extraction Level General Arrangement



Source: LUCKAR05E-1700-MIN -DDD-J900_Rev C (JDS 2023)

16.7.4.1 Level Selection

The Extraction Level elevation was selected through a series of Arena simulations (SRK, 2020). The purpose of the Arena simulation was to battery test the extraction level of the UGP and find bottlenecks, quantify equipment requirements, and maximum capacity of the system. The simulation incorporated the collective availability and throughput of all points of material movement from the drawpoints through to shaft discharge on surface including drawbells, Load Haul Dump (LHD) machines, breaking equipment, crushers, conveyors, bins, and skips.

310 masl was selected as the base extraction level in this exercise, as it had been previously identified as the most profitable extraction level in 2019 PCBC Footprint Finder evaluations (JDS, 2019). The Arena simulations stepped the extraction level downwards in 10 m increments to the

bottom of the indicated resource (250 masl) and tested the system capacity using the following constraints:

- Drawpoint spacing 28 m;
- Crosscut spacing 31.5 m;
- Ore density 2.9 t/m^3;
- LHD count cannot exceed number of extraction drives;
- LHD hours cannot exceed 4,600 per annum;
- Non autonomous LHD operation – machines stop at shift change and during blasts;
- Non-electric LHD operation – machines stop every 7 hours for fueling;
- Planned maintenance of 12 hours every 250 hours; and
- Daily Pit stops of 40 minutes.

The results of the Arena simulation suggested that the target production rate of 2.7 Mtpa could be achieved as low as 280 masl, however, a decision was made to hold the extraction level at 310 given the simulation did not take into account geotechnical, hydrogeological, or operating project risk. Given the 310 masl offers 35% more drawpoints than 280 masl, it is anticipated that this 35% surplus will account for these risks.

16.7.4.2 Drawbells/Troughs

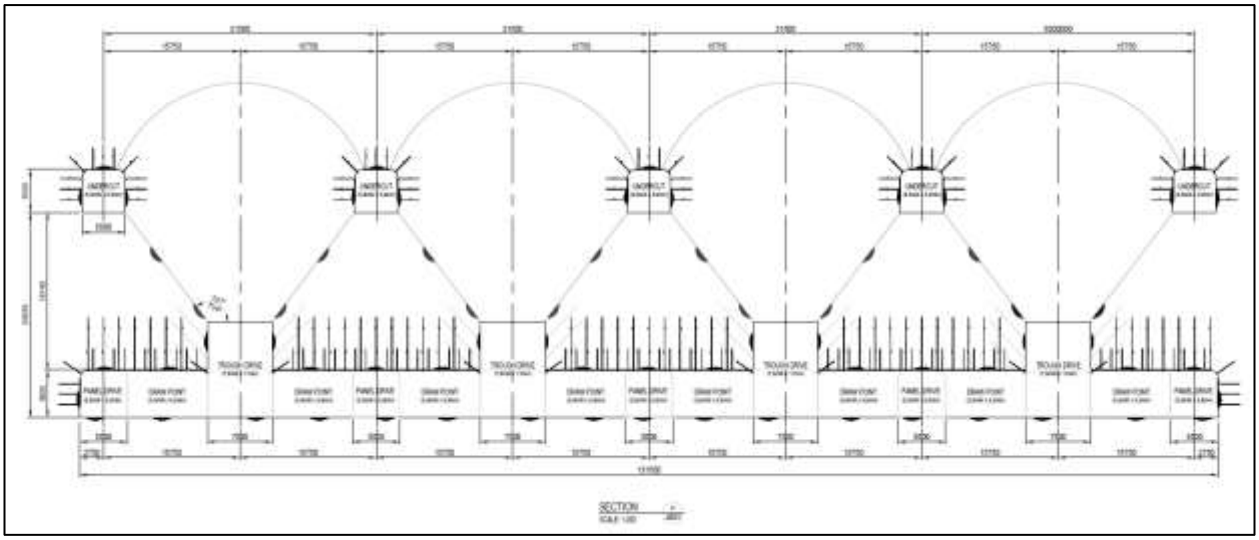
Drawbells will be developed in a trough style manor (TDR009, 2021). Trough drives will be developed in between the Extraction Drives at 7.5 m W x 11 m H the full length of the drive. Drawbells will be drilled from two undercut panels above the Extraction level which form the peak of the Apex Pillar. The Drawbells will be mined in retreat, blasting the contents to the trough drive below. Figure 16-46 illustrates the Drawbell arrangement.

Trough style drawbells are different from typical caving layouts in that there are no secondary Apex Pillars. There are fewer, larger, drawbells which feed multiple crosscutting drawpoints.

Trough style drawbells are not common given their association with caving, and caving's association with weak ore. Typical drawpoints utilize secondary (minor) Apex Pillars between each drawpoint to help distribute the weight of the caved material on the pillars and manage stress. Trough style drawbells require a competent host rock free of jointing or faults that can withstand a high degree of loading. The South Lobe is expected to meet these conditions.

Benefits of the trough style drawpoints are primarily associated with the time and cost to bring a mine into production. Approximately 10x fewer slot cuts are required in the trough style arrangement, and blasting the drawbells in reverse is akin to a longitudinal retreat stoping operation. Confidence in both geomechanical modelling and drill and blast execution becomes much higher in this method, however, as failure of one Apex Pillar could truncate production by as much as 20%.

Figure 16-46: Trough General Arrangement

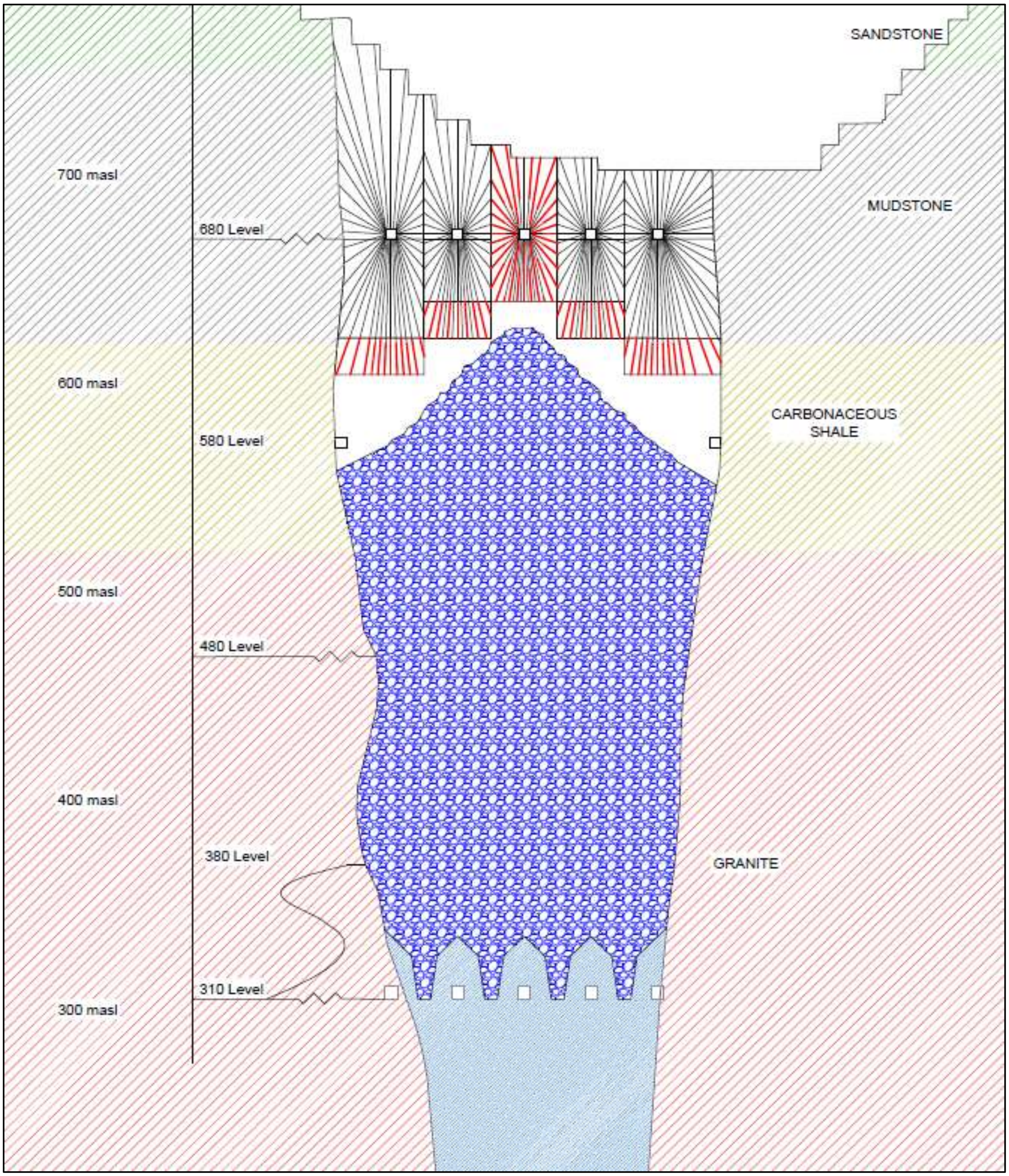


Source: LUCKAR05E-1700-MIN -DDD-J903_Rev C (JDS 2023)

16.7.5 Crown Pillars and Sill Pillars

Crown and sill pillars are designed to be a minimum 30 m, exceeding by small margin the 25 m minimum outlined in Section 16.3. To avoid the requirement of maintaining access to the pit bottom long after pit closure, it has been planned to drill the crown pillar from the 680 Horizon using Sandvik DU311TK long hole drills. It will be important to manage pit sumps through to the point of crown pillar recovery to avoid instances of inrush. Detailed crown pillar drill and blast plans shall be prepared closer to the time of recovery, however, the general approach involves a central glory hole to the shaft floor followed by a few mass blasts to be conducted in unison with the 670 sill pillar wreckage.

Figure 16-47: Crown Pillar Opening



Source: JDS (2023)

16.7.6 Drilling Horizons

The first five (5) levels of the mine will serve primarily as access for the drilling and charging of long hole stopes. These levels include:

- 680 L;
- 580 L;
- 480 L;
- 380 L; and
- 340 L.

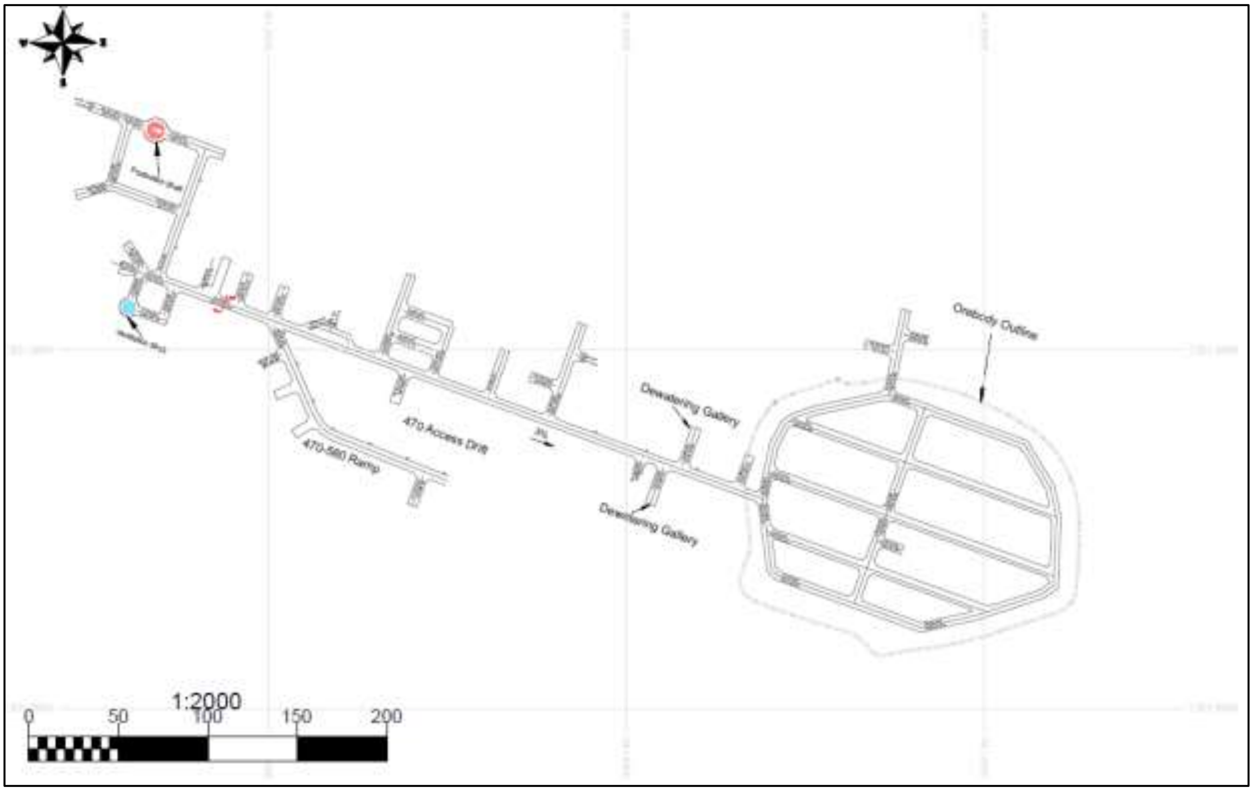
Drilling Horizons will be developed at 5.0 m W x 5.0 m H unless otherwise specified for specific infrastructure and accessed by either shaft station or ramp from another level.

Drilling Horizons will be equipped with the basic infrastructure required to operate long hole drilling and charging equipment, including but not limited to satellite shops, magazines, sumps and pump stations, mine power centers, refuge chambers, ventilation infrastructure, and definition/depressurization drill bays. Where drill horizons are connected by ramp to other levels, infrastructure such as shops and magazines will be shared.

Within the ore body, 5.0 m W x 5.0 m H drill panels will be excavated on 30 m spacing across the South Lobe. A central crosscut will be driven perpendicular to these panels to serve as access for the slot raise required to start stoping. An additional drift will be driven inset from the circumference of the South Lobe to connect each drive together and provide access to the far end of the drill horizon once the central slot has been excavated. Drill horizons typically have four to five parallel drill panels, one perimeter drive, and one central crosscut.

Figure 16-48 illustrates a typical drill horizon.

Figure 16-48: 480 Drill Horizon Plan View



Source: JDS (2023)

Waste rock generated through development of the 680 L, 580 L, and 480 L drill horizons will be trammed to a 2.1 m diameter waste pass located near the shafts. Tips will be equipped with a static grizzly and mobile rock breaker to reduce oversize to a minimum 300 mm passing. Muck passes will be interlinked via finger raise and ultimately report to one of two fine ore bin collars at 335 Station. A bulkhead and transfer conveyor on 335 Station will control feed into the fine ore bins such that the bins may campaign ore and waste containment. Waste passes will be developed by raisebore. Requirements for raise support or grouting and will be subject to evaluation closer to execution.

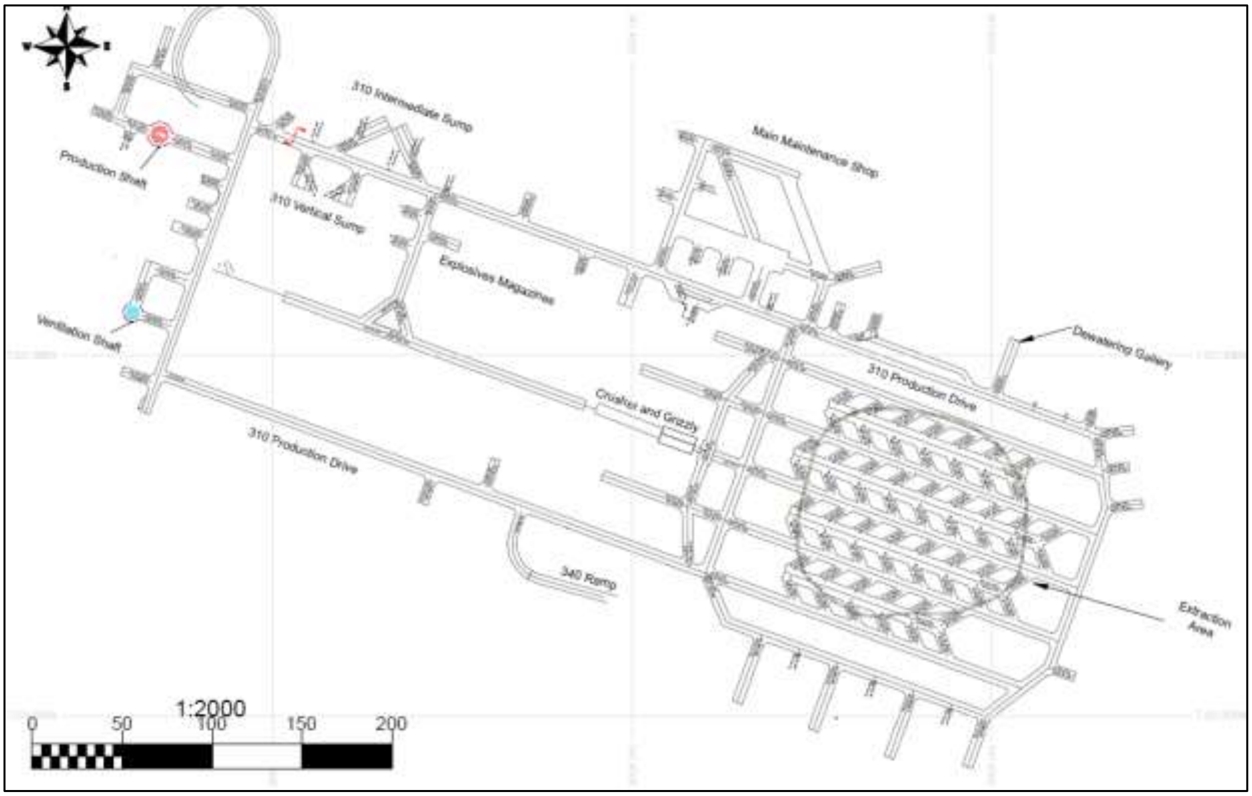
Fresh air will be supplied to the drill horizons by a FAR, 4.0 m in diameter, connected to surface. The FAR will connect to the 680 L, 580 L, 480 L, and 380 L drill horizons by access drifts, each equipped with regulators to control the ventilation airflow entering the level.

Drill Horizon development will be sequenced from bottom-up, in theme with the direction of mining. The first drill horizon to be developed is the 340 L undercut, and the last is the 680 L production horizon. Drill Horizons will be temporary in nature and closed off once all drill and blast activities have been completed. Station infrastructure including sumps and pump stations will remain in service for the life of mine.

16.7.7 Extraction Horizon

The extraction level is located at 310 masl (L) and is accessed from the 310 L P/S shaft station. Figure 16-49 shows a plan view of the 310 L.

Figure 16-49: 310 L Plan View



Source: JDS (2023)

The 310 L will remain active for the life of mine and provide access to the following infrastructure:

- Drawpoints and Primary Tip;
- Crusher Conveyor;
- Workshop and Warehouse;
- Magazine;
- Permanent Refuge;

- Primary Sumps;
- Ramp to 340 L and 380 L Drill Horizon; and
- Ramp to 285 L Services Horizon.

The 310 L will be accessed via the P/S. Twin drives will be developed towards the South Lobe with crosscut connections before and after the ore body to provide ventilation circuits and 360 degree access around the extraction drives. As the 310 L receives the highest concentration of fresh air the twin drives will help to split flows and reduce air velocities which would otherwise require oversized excavations. The Northern Drive will house the bulk of the UG infrastructure including sumps, workshops, magazines, refuge, and access to crusher and conveyor. The Southern Drive will contain minimal infrastructure to allow for expedited development towards the ore body, a dedicated haulage route to minimize pedestrian and vehicular traffic interaction, and ramp access to upper drill horizons.

Development on the Extraction Level will be a minimum of 5.5 m W x 5.5 m H to accommodate the planned 21 t drawpoint mucking loader, unless otherwise specified for infrastructure needs.

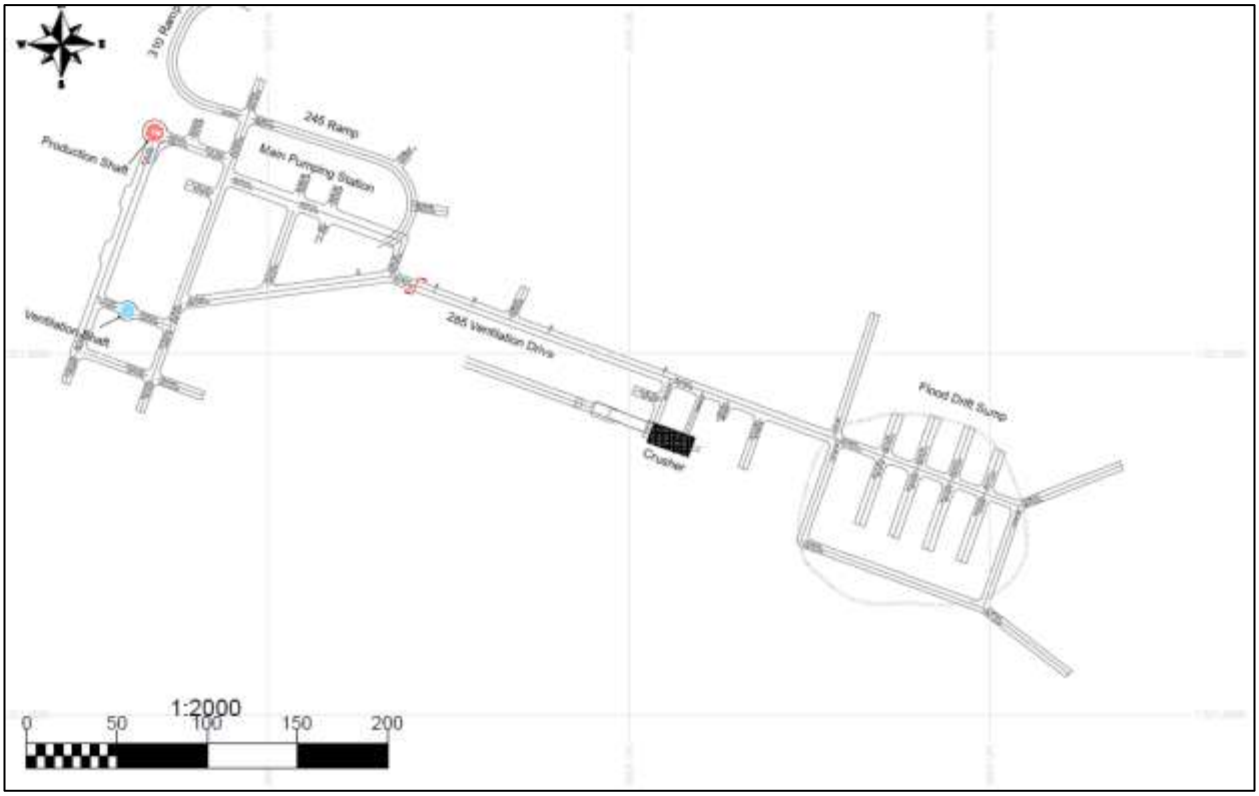
Development will grade towards the shafts up to the entry of the Extraction perimeter drive in an effort to report all water produced during development back towards the shaft station sump. Within the Extraction perimeter drive development will grade to the East, away from the shafts, and towards a pair of sumps dedicated to managing groundwater inflows produced by the cave.

Waste rock generated through the development of the 310 L will be trammed towards the V/S and tipped down a rock pass equipped with static grizzly and serviced by mobile breaker. This pass will report to the skip loading station where another LHD will rehandle material onto the tail end of the skip loading conveyor.

16.7.8 Services Horizon

The Services Horizon is located at 285 masl (L) and is accessed from the 285 L P/S shaft station. Figure 16-50 shows a plan view of the 285 L.

Figure 16-50: 285 L Plan View



Source: JDS (2023)

The 285 L will remain active for the life of mine and provide access to the following infrastructure:

- Skip Loading Conveyors and chutes;
- Primary Pumping Station;
- UG Crusher;
- Flood Chamber;
- Ramp to P/S Bottom; and
- Primary Return Air Way.

The Services Horizon will be air locked, separating the P/S (fresh air) from the V/S (return air), and equipped with double doors for vehicle access when required.

The 285 L will be accessed via the P/S, as well as by ramp from the 310 L. A single drive will extend from the shaft station towards, underneath, and East of the 310 L extraction area. Ventilation raises will be driven from the 310 L to the 285 L along this drive to provide key ventilation return circuits. This drive will serve as access to the main UG pump station and crusher, further described in Section 16.8.

Development on the Services Horizon will be a minimum of 5.0 m W x 5.0 m H unless otherwise specified by infrastructure requirements, with the primary return airway requiring a larger 6.0 m W x 6.0 m H to reduce air velocities.

16.7.9 Flood Chamber

Development will grade towards the shafts up to the entry of the crusher chamber in an effort to report all water produced during development back towards the shaft station sump. Beyond this point development will grade towards the ore body and into a flood chamber, sized for 26,000 m^3 capacity. This flood chamber, further described in Section 16.8, will serve dual purpose as the primary return air circuit for the mine. The flood chamber will be developed as twin drives, one upper, and one lower, such that the lower drive will flood before the upper without severing ventilation circuits. Crosscuts will be developed from the twin drives as needed to expand upon flood storage requirements. The flood drives will receive water as overflow from sumps located East of the drawpoints, reporting down one of two ventilation raises to the flood chamber. Submersible sumps will pump water from the flood chamber as needed back into the overflowing sumps until the dewatering system capacity has caught up. A hydrostatic flood door will be installed prior to the flood chamber to protect the mine infrastructure on the Services Horizon in the event of an extreme flooding event.

Access to the flood chamber should be required only for the periodic mucking of slimes. As it will be the hottest and wettest area of the mine access will be limited to those trained specifically for the area.

16.7.10 Crusher and Conveyor Levels

The crusher is located below the 310 L extraction area and will have two primary points of access:

1) Bottom entry via the 285 L return air drive; and

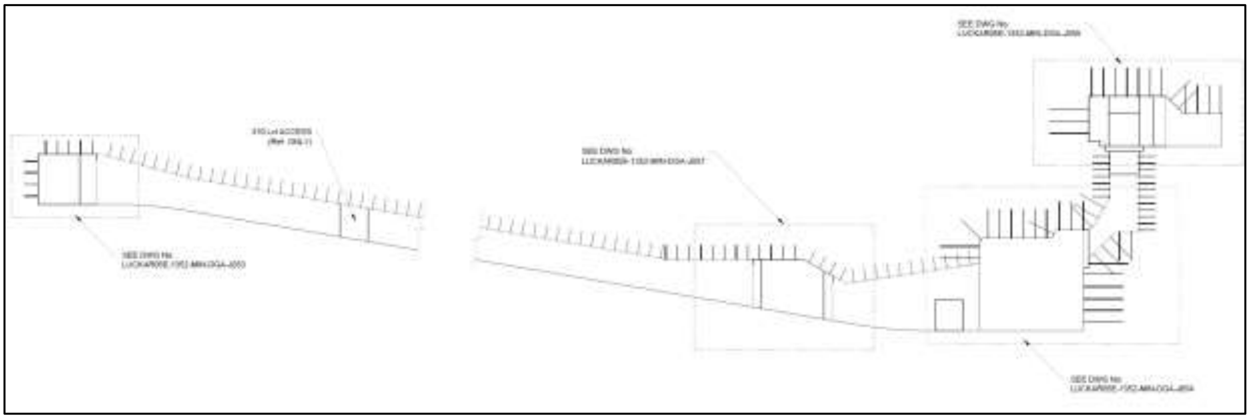
2) Top entry via the conveyor drive, which has an access point on the 310 L.

The crusher chamber will be approximately 9 m W x 18 m H x 20 m L. A 6 m W x 6 m H conveyor drive of maximum 17% gradient will connect the crusher chamber and the 335 L together. The conveyor drive will be used initially as an attack ramp to excavate the crusher chamber in lifts using standard drill and blast equipment. The conveyor drive will have a mid-point access on the 310 L, which will provide all services to the crusher chamber.

Bottom entry to the crusher chamber from the 285 L return air drive will provide access for construction and maintenance equipment. The conveyor drive will be wide enough to accommodate light equipment access.

The crusher and conveyor will be naturally ventilated from the 285 L return air drive. Fresh air will be forced down the conveyor drive via fan and ducting located on the 310 L.

Figure 16-51: UG Crusher and Conveyor Excavation Layout



Source: LUCKAR05E-1352-MIN-SEC-J052, Material Handling Ground Support Overview (JDS 2023)

16.7.11 Raises

Internal intake and exhaust raises will be used to bring fresh air into the extraction area and exhaust air towards the V/S. This will ensure a constant supply of fresh air to the main working area. Raises greater than 30 m will be driven by a raise bore machine, and those less will be done with a long hole drill.

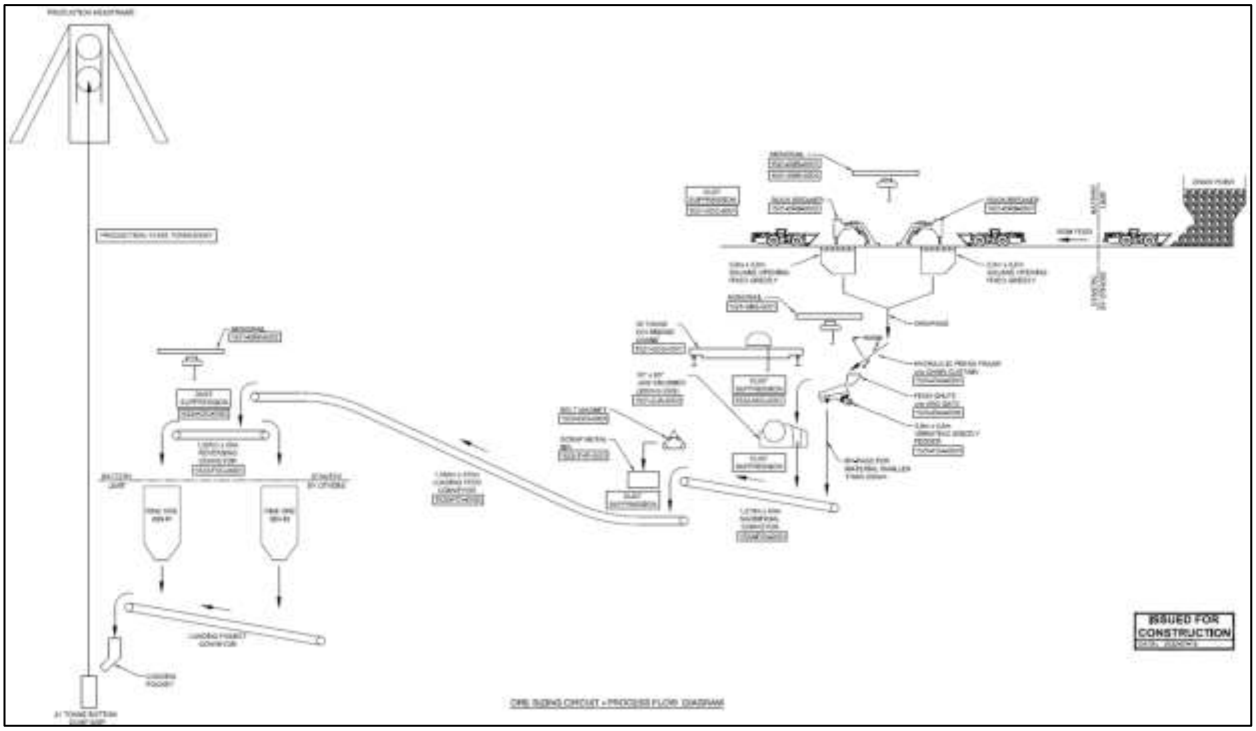
A raisebore machine will drive 3.0 m diameter raises within the kimberlite to serve as production slot raises, development muck passes, and fresh air ventilation between working levels. Raises will be driven in multiple sections from the main extraction level to the topmost drill horizon, and to surface within the OP.

16.8 Mine Services

16.8.1 Comminution Circuit

The comminution circuit consists of single stage crushing and UG conveying to a double drum skip hoisting system. Figure 16-52 illustrates the UG material flow from drawpoints to the surface.

Figure 16-52: Comminution Circuit Process Flow Diagram



Source: Stantec (2023)

Production targets are for 441 t/h or 7,500 t/d of material, however the crushing and conveying circuit has been designed at a higher capacity to balance fluctuations in LHD feed and fine ore bin levels. Key design criteria for the comminution system is outlined below in Table 16-9.

Table 16-9: Comminution System Key Design Criteria

Design Criteria	Units	Parameter
Plant Availability	%	70
Operating Days per Year	days	360
Crusher Operating Hours per Day	hours	16.8
Crusher Feed Top Size	mm	800
Crusher Throughput Capacity	t/d	10,805
	t/hr	643
Crusher Product Size	P_{80} mm	200

Source: JDS (2023)

Production LHDs will muck ore directly from drawpoints to a central, double wide, three-sided UG grizzly. Material will be dumped onto an 800 mm static grizzly above an ore pass. Oversized material from the static grizzly will be reduced in size by a single BTI MRH T 20/25 BXR50 teleremote-ready rockbreaker. During initial ramp up an operator will control the rockbreaker from a local control booth; but teleremote operations and the future expansion of a second rockbreaker is envisioned during operations. Difficult to break material can be removed via the LHD or rockbreaker and taken to a remuck for secondary breakage.

The ore pass is designed with flat bases to encourage rock on rock wear across a retained bed of broken ore, and to minimize the maintenance of steel liners. At the bottom of the ore pass, a chute with chain press frame and arc gate will control material flow onto the Astec GBEX 2000 mm x 4400 mm vibrating grizzly feeder. As the material advances along the vibrating grizzly feeder, a series of tapered grizzly-bars will allow for undersized -200 mm material to bypass the crusher and feed directly onto the sacrificial conveyor. Oversized material will pass over the end of the vibrating grizzly feeder deck directly into a Telsmith 1,270 mm x 1,524 mm (50" x 60") 6 Piece Single Toggle Jaw Crusher with an installed power of 250 kW.

The primary crushing stage will produce a target P80 of 200 mm at a crusher closed side setting (CSS) of 180 mm, for the sacrificial conveyor. The 1,270 mm x 1,524 mm jaw crusher selected for the Crushing and Conveyor System is oversized and selected based on the maximum feed size, not throughput targets. Similarly, the vibrating grizzly feeder has an oversized pan in order to keep material free flowing, the equipment selection is based on the maximum feed size, exceeding throughput targets.

The 50 m long, 1200 mm wide sacrificial conveyor will be equipped with a belt magnet to retrieve rock bolts and other metalliferous material that may cause damage to the main conveyor and hoisting system. Scrap metal will be pulled aside and disposed of. The sacrificial conveyor will transfer material onto a 290 m long, 1050 mm wide loading feed conveyor with an installed power of 160 kW required. A 30 m long, 1050 mm wide reversing conveyor will discharge material, transferred from the loading feed conveyor, into the top of one of two 50 m tall fine ore bins.

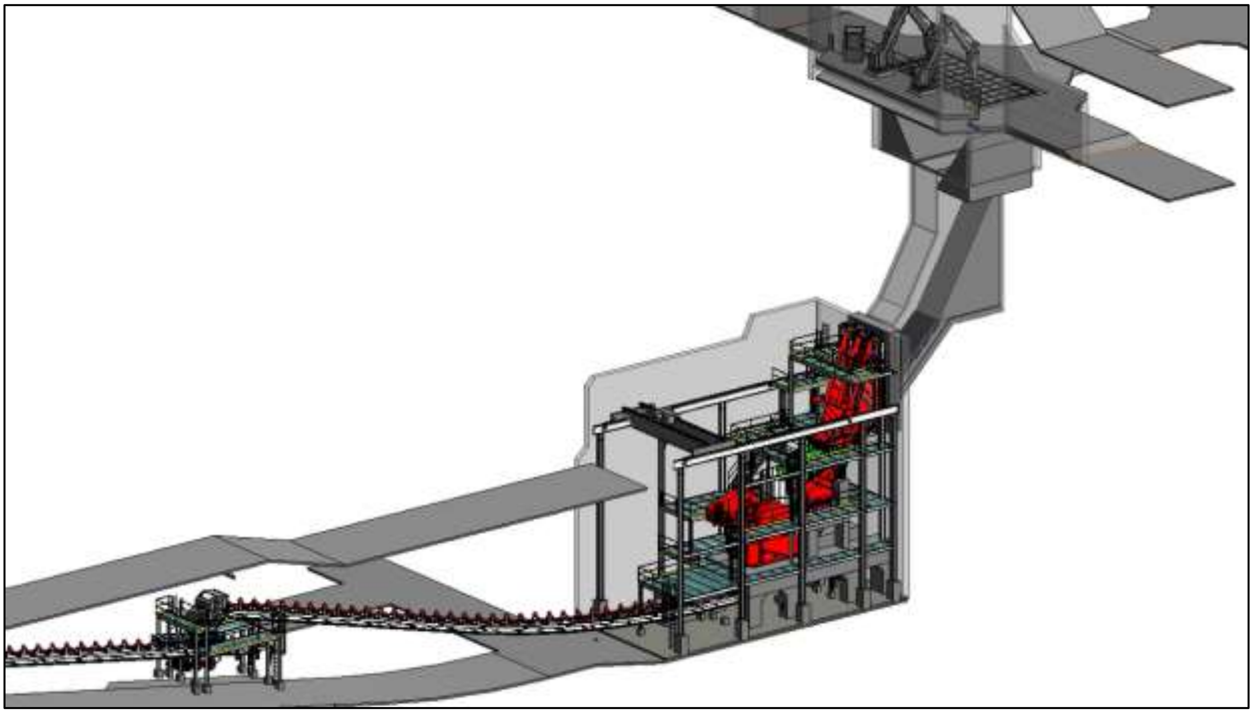
Chutes at the bottom of the fine ore bins will feed skip loading conveyors in a controlled fashion to meet skip demand. 21 t skips will hoist ore to surface, and surface haul trucks will transport material from the shafts to the existing mill for final processing.

The entire crushing area will be covered by a 35 t overhead bridge crane, capable of lifting the entire swing jaw assembly in a single lift. This crane will service the vibrating feeder and associated chute-work, the jaw crusher and associated chute work, the sacrificial conveyor, and the belt magnet. A series of D-rings will be installed over the feed chute to assist in the maintenance of the feed chute, press frame, and control chains.

Atomizers will be used to mist water for dust suppression from a manifold bar by the static grizzly, as well as from nozzles at the top of transfer chutes. Dust collected by the small water particles will fall onto the moving bed of material, rather than implementing a system of dust collectors.

The layout of the Crusher and Conveyor System is outlined below in Figure 16-53.

Figure 16-53: Crusher and Conveyor System



Source: JDS (2023)

16.8.2 Dewatering

16.8.2.1 UG Water Management Requirements

During the construction period and in the initial years of operations, two sources of water inflows require management UG:

1) Groundwater Inflows - Estimates have been performed utilizing a groundwater model as outlined in Section 16.4; and

2) Mine Service Water – A maximum consumption rate of 340 m^3/day has been estimated to support mobile equipment, however a rule of thumb of 1200 m^3/day of available supply has been utilized for water management requirements to incorporate some contingency into the design.

Once a connection is made between the OP and the UG workings in Q1 2030, the UG workings will need to manage inflows from precipitation. The OP will provide a catchment area for any precipitation and/or groundwater entering the OP. These volumes of water will have the opportunity to flow into the UG mine from the surface of the OP and to travel down via fractures or lithological contacts to the open cavities of the UG mine. Precipitation is estimated into rainfall and stormwater volumes.

Rainfall volumes are generally low in the region. Stormwater volumes, however, can be considerable under a very short period of time, during extreme storm events. Failure of the UG mine dewatering infrastructure to have sufficient capacity would result in the flooding of the UG mine, which would have significant consequences.

Key input parameters are outlined below in Table 16-10.

Table 16-10: Precipitation Estimates

Item	Units	Value
Catchment Area		
Pit Catchment Area	m^2	413,696
Runoff Coefficient	-	0.85
Rainfall		
Typical Annual Rainfall	mm	372
Month of Highest Rainfall (January)	mm	94.2
Stormwater		
Storm Event	-	1:100
1:100 Storm Event	mm	258 mm over 4 days
UG Storage Capacity	m^3	26,000

Source: JDS (2023)

Based on the inflows estimated from the three primary sources, the estimate peak inflows from each elevation are outlined below in Table 16-11. 2030 represents the year of the highest daily inflows, after which estimate inflows begin to delivery over the life of mine.

Table 16-11: Peak Daily Groundwater Inflows by Level (m^3/day)

Date	Peak (each)	Avg (each)	2023	2024	2025	2026	2027	2028	2029	2030	2031
Groundwater Inflows											
670 Shaft Water Ring	6	6	6	6	6	6	6	6	6	6	6
285 Shaft Water Ring	6	6	-	6	6	6	6	6	6	6	6
670	476	297	25	45	41	130	219	366	376	476	471

Date	Peak (each)	Avg (each)	2023	2024	2025	2026	2027	2028	2029	2030	2031
580	333	156	4	12	11	290	333	309	257	239	192
470	2,731	1,257	-	425	389	1,157	2,731	2,305	1,790	1,537	1,369
380	1,823	1,034	-	34	104	298	1,222	1,823	1,588	1,420	1,301
340	1,054	502	-	0	53	999	1,054	641	525	494	473
310	5,274	2,650	-	197	562	5,274	4,591	3,698	3,092	2,898	2,756
285	392	268	-	3	43	179	170	392	374	353	338
Drawbells	629	432	-	-	-	-	37	117	208	320	399
Mine Service Water											
670	219	126	9	12	-	-	-	91	203	219	219
570	219	190	-	-	-	-	136	214	219	-	-
470	219	147	-	14	-	-	219	209	-	-	-
380	155	83	-	-	-	27	155	66	-	-	-
340	112	81	-	-	-	49	112	-	-	-	-
310	132	51	-	5	19	132	74	49	49	49	49
285	82	41	-	3	82	43	-	37	-	-	-
Contingency	1,142	1,036	-	-	-	1,142	1,104	966	985	981	-
Precipitation											
Precipitation	1,069	1,045	-	-	-	-	-	-	-	813	1069
Storm Water (1:100)	16,181	16,181	-	-	-	-	-	-	-	16,181	16,181
Total	**24,810**	**16,791**	**44**	**702**	**1,159**	**8,991**	**10,331**	**10,258**	**9,234**	**24,810**	**24,745**

Source: JDS (2023)

16.8.2.2 Dewatering System

The KDM UGP Dewatering System is a single lift dirty water pumping system.

As outlined in Section 16.4, the Dewatering System is challenged by the high-temperature, corrosive groundwater inflows. The selection of main pumps is further challenged by the abrasive nature of the water due to dirty water pumping. A positive displacement pumping solution was the only feasible option to balance these requirements, as the pumps are robust, relatively simple to maintain, and a proven technology in the region.

With the dirty water pumping system, larger particles (>8 mm) will need to be removed from the system, but in general, the settling of slimes UG is to be mitigated. There are limited facilities UG to manage slimes, the system is designed to transport slimes to surface where they are managed at a surface facility.

Key specifications for the Dewatering System are outlined below in Table 16-12.

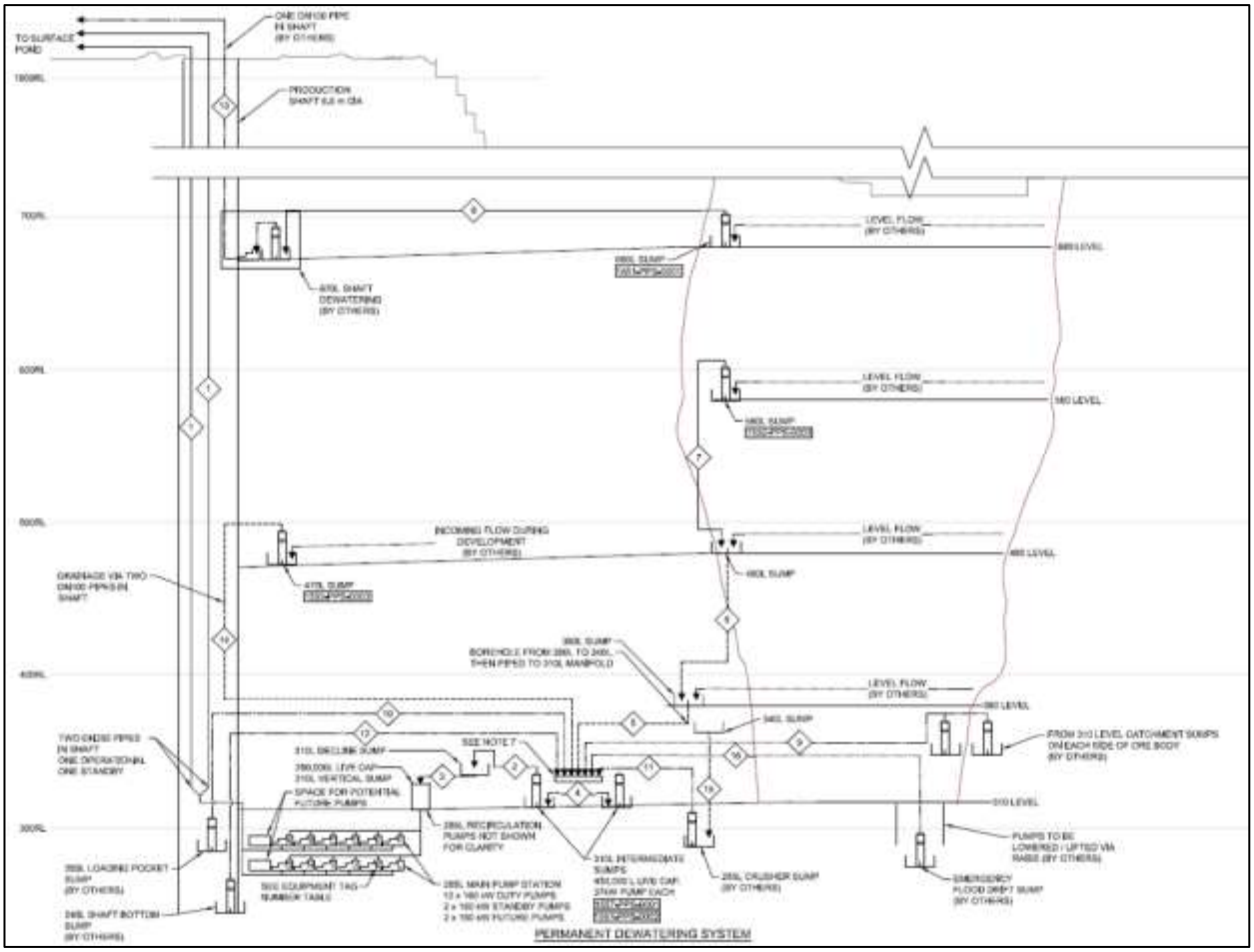
Table 16-12: Key Dewatering System Specifications

Item	Units	Value
General Conditions		
Fluid Type	-	Dirty Water
System Flow Rate (Design)	m^3/hr	500
Estimated Discharge Pressure	MPa	8.7
Rated Duty - All Pumps	hours/day	24
Estimated Average Running Time – All Pumps	hours/day	20
Emergency Running Time – All Pumps	Hours/day	22
Solids Concentration - Maximum	% by weight	15%
Solids Concentration - Average	% by weight	1%
Fluid Density - Maximum	kg/m^3	1100
Maximum Particle Size	mm	8
Temperature	°C	50
Water Chemistry		
Water pH	-	7.26
Electrolytic Conductivity (EC)	MicroS/cm	37,400
Total Dissolved Solids	mg/l	25,000 – 33,000
Chloride, Cl	mg/l	10,000 – 20,000

Source: JDS (2023)

The Dewatering System consists of various local sumps, level sumps, pipelines, and pumps throughout the mine. The Dewatering System Process Flow Diagram is outlined below in Figure 16-54.

Figure 16-54: Dewatering Process Flow Diagram



Source: Stantec (2023)

The Dewatering System cascades and directs the majority of water to a UG Main Pumping Station which consist of 6 components:

1) 670 L Pump Station;
2) Level Sumps;
3) 310 L Intermediate Sump;
4) Vertical Sump;
5) 285 L Main Pump Room; and
6) 285 L Stormwater Pump Room.

16.8.2.3 670 L Pump Station

During shaft sinking and station development at the 670 L, two Scamont FXG 37 kW pumps will be installed for future use as a 670 L pump station. Initially these pumps will capture shaft inflows to the water rings, as well as station development inflows, however in the future, additional inflows will be captured as the development contractor progresses towards the orebody. This pump station is strategically positioned to split the water inflows with varying chemistry, inflows to this pump station are expected to be slightly saline, can potentially be utilized UG for service water, and can be utilized at surface without treatment. All dewatering systems below this pump station will be managing highly saline water.

16.8.2.4 Level Sumps

A series of typical decline and borehole sumps will be used throughout the mine to cascade water to the 310 L Intermediate Sump. Decline sumps will contain a catwalk and submersible pump, while borehole sumps will connect sumps between levels when spatially feasible. In general, mobilizing all particles smaller than 8 mm will be a target in the dewatering system to limit slimes management on each level.

16.8.2.5 310 L Intermediate Sump

All mine water will report to a pipe manifold at the 310 L intermediate sumps. From here, submersible pump(s) shall move the water to the top of the Vertical Sump. The intermediate sump will be used to settle the larger particles and intercept debris from the run of mine water in the final, permanent dewatering configuration for the mine. The operation of the 310 L intermediate sumps shall be such that all incoming water will be directed to one of the two sumps via valving on the incoming pipes, with the other sump being cleaned or ready on standby. The intermediate sump will also allow for temporary storage capacity during system downtime.

16.8.2.6 Vertical Sump

Two vertical sumps shall be located between the 310 L Intermediate Sump and the Main Pump Room. The vertical sumps will be a 4 m x 4 m excavation spanning from the 310 L down to the 285 L with a vertical hydrostatic bulkhead located at the bottom, separating the sumps from the main pump station on 285 L. The top of each vertical sump will have a short, traditional decline sump which will act to settle the larger particles, and trap debris that enters the sumps prior to cascading into the vertical sump.

A vertical sump will be used to feed the pumps via a flooded suction line, eliminating the requirement for feed pumps, instrumentation, and the programming required to provide adequate net positive suction head. The capacity of the vertical sump will also allow for fluctuations in groundwater inflows and pumping rates.

16.8.2.7 285 L Main Pump Room

Scamont SP200 160 kW Positive Displacement Pumps have been selected as the Main Pumps and are to be installed at the 285 L Main Pump Room. These pumps will have the wet end upgraded to duplex stainless steel to prevent against corrosion. The pumps will be fed from a common suction header installed through the hydrostatic bulkhead at the bottom of the vertical sump. The common suction header will split to two banks of pumps in order to limit the length of the suction line. In each bank of pumps, 6 pumps are to be installed, 5 duty and 1 standby pump, with room of expansion. 12 pumps in total are in the design to meet target inflow rates.

The 285 L Main Pump Station will deliver water to surface through a 250 NB pipe in the P/S. A second 250 NB pipe will be installed as a standby and may be used in the event of stormwater pumping, if required. The surface water management facilities are outlined in Section 18.2.

A dedicated electrical room will be required for the Main Pumping Station and will be positioned nearby the 285 L Main Pump Room. The control of the main pumps should be automated, however, the mine intends to operate with full time pump operators, monitoring and maintain the main pumps frequently.

16.8.2.8 285 L Stormwater Pump Room

To manage potential stormwater inflows in early 2030, an expansion to the 285 L Main Pump Room is planned. A preliminary selection of Scamont GSB 150 1260 kW self-balancing multi-stage pumps in a 2 duty and 1 standby arrangement has been made. These pumps require a degritting system upstream to remove particle sizes above 125 microns (µm), however dirty water pumping and slimes are possible through these pumps with a much smaller footprint than the comparable positive displacement pumps. Upgrading of the wet end for duplex stainless steel is very costly, since the pumps will be used primarily to manage stormwater inflows, standard cast iron construction will be sufficient. The pumps will be placed nearby the positive displacement pumps in order to utilize the same vertical sumps as well as the supporting electrical, controls and instrumentation infrastructure.

16.8.2.9 Dewatering Piping

Dewatering (dirty mine water) is primarily a combination of groundwater inflows, excess service water, and precipitation. As groundwater inflow within the mine have high salinity, HDPE piping is preferred to prevent corrosion. In general, dewatering (dirty mine water) is cascaded at low pressures to a main pumping station on the 285 L using HDPE piping. At the high temperatures expected from the groundwater inflows, the HDPE pressure rates will be derated significantly. In the 285 L pumping stations, dirty mine water is pumped in a single lift system out of the mine; this high-pressure pumping requires a specifically designed high pressure carbon steel piping specification, with some specialized duplex stainless steel fittings and spools.

16.8.2.10 Water Disposal

From the UG operations water will be pumped to a surface water management facility, which is outlined in Section 17.4.8 and 18.2.

16.8.3 Mine Ventilation

16.8.3.1 Ventilation

The ventilation system for KDM UGP is based on a planned production rate of 2.7 Mtpa, the development and production schedules, and the respective airflow demand estimates over the LOM.

The P/S serves as the sole fresh air intake for the mine. The V/S serves as primary exhaust. Crosscuts between shafts on working levels will establish ventilation circuits from which on-level development fans, regulators, and bulkheads will direct fresh air to working locations. Inter-level raises will provide flow-through ventilation to quickly exhaust production blast gasses where possible. A raise connecting the levels to the OP bottom will act as secondary exhaust. Figure 16-55 depicts the primary ventilation network of the mine during steady-state production (Year 2029 through end of LOM).

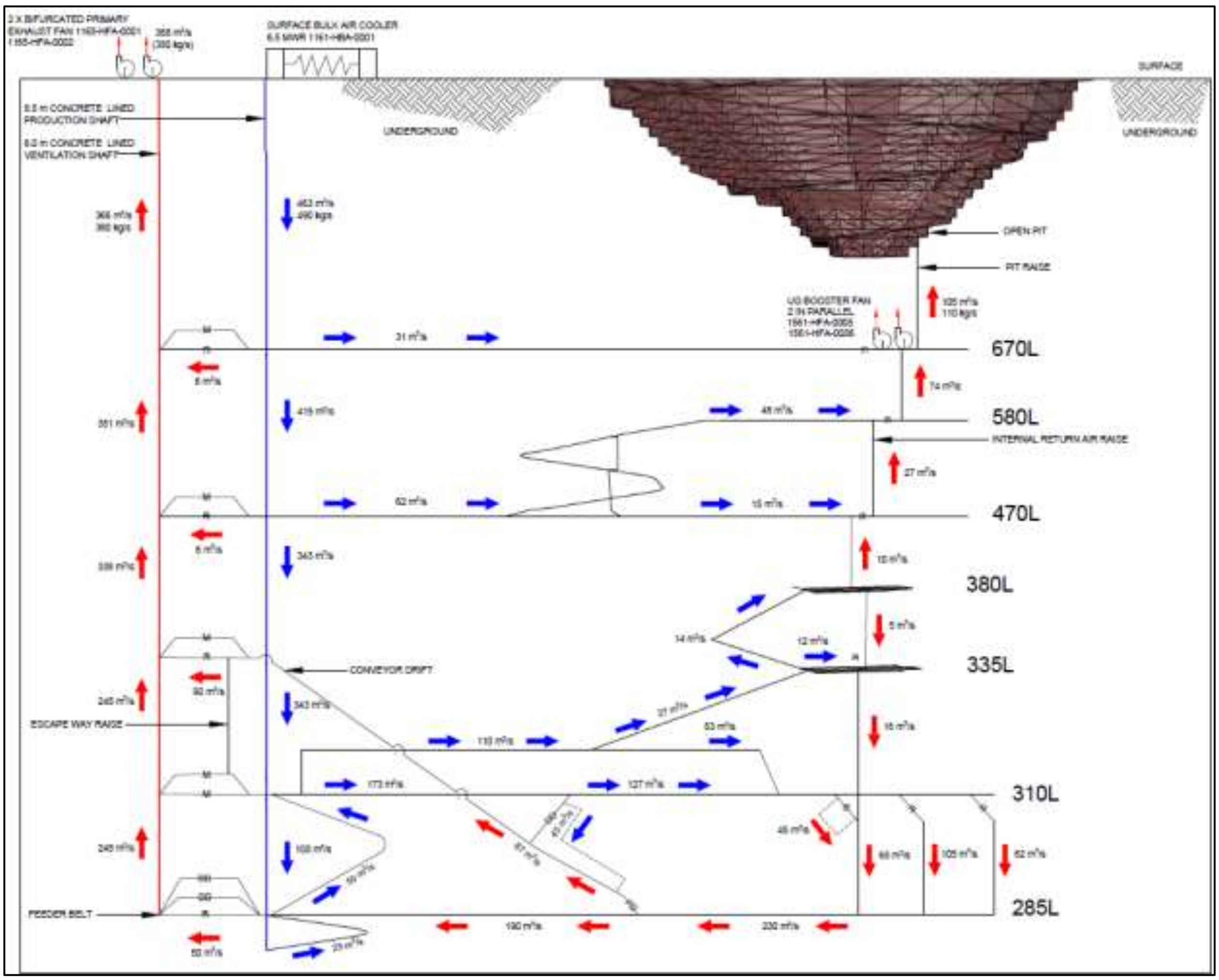
The mine ventilation will operate as a pull system with one fresh air source and two return air routes. Twin surface mounted fans will draw fresh air down the P/S, through the lower mine levels, and back to surface via the V/S. A second set of twin UG booster fans will draw fresh air from the P/S across upper mine levels and out an in-pit ventilation raise.

Peak airflow demand for the UGP is 435 m^3/s based on ventilation requirements of planned UG diesel-powered equipment, personnel, fixed infrastructure, and development schedule. Airflow distribution varies with time and schedule as the mining activities progress in each level. The twin main fan and twin booster fan installations have a combined airflow capacity of 510 m^3/s to allow for the management of the flow through working areas, fixed facilities, and haulage airways, and to account for auto-compression of the airflow at depth. Peak airflow demand compares well against benchmarked caving mines of similar size.

The primary design criteria for supplying air to the diesel-powered equipment (0.05 m^3/s per installed kilowatt of rated diesel engine power) is derived from the Botswanan *Mines, Quarries, Works and Machinery Act* (MQMA) regulation108(f). Air velocities UG are kept within general mining industry standards. Additionally, areas that represent an elevated ventilation risk, such as crusher chamber, main pump rooms, preventive maintenance bays, etc., are planned to direct exhaust straight to a return airway to prevent the entrainment of smoke, heat, and other contaminants into the main ventilation system in the event of a fire in one of those facilities.

The estimated base ventilation demand for the KDM UGP of 435 m^3/s is based on a build-up of the individual active zones, stopes, and other areas (bottom-up view), and on the diesel equipment fleet requirements (top-down view). This value falls within the ±15% confidence interval of other estimates, including peer comparison benchmark method (370 – 501 m^3/s), baseline comparison (394 – 429 m^3/s), and third-party engineering design.

Figure 16-55: KDM UGP Ventilation Long-Section



Source: LUCKAR05E-1500-MIN-PFD-J118 (Ventilation Process Flow Diagram)

The mine's total installed exhausting capacity will be 510 m^3/s, or 110% of the demand, and accomplished through two primary fan installations. The first installation is at the collar of V/S, where two parallel 750 kW fans are planned to exhaust up to 366 m^3/s at a pressure of 1.7 kilopascal (kPa). The second installation is at the bottom access of pit raise on 670 L where two parallel 45 kW fans are planned to exhaust up to 110 m^3/s at a pressure of 0.5 kPa. The combination of both fan arrangements will draw 463 m^3/s from the mine, which after compensation for auto compression will achieve an UG flow rate of 435 m^3/s.

Primary ventilation support is achieved using bulkheads, conventional airflow regulators, and airlocks to allow personnel and equipment to move from circuit to circuit without disrupting the overall ventilation system. Ventilation distribution management is based on using pressure drop regulators (passive controls) rather than pressure boosting fans (active controls). Additional primary support is achieved using auxiliary ventilation fans and ventilation bags to deliver air to the end of development and production headings. These systems are sized to support the anticipated equipment and personnel requirements based on the requirements in Botswanan MQMA.

Ventilation equipment will be manually operated without automation controls. Fans will be locally controlled from the MCC, and regulators will be manually opened and closed. Gas monitoring equipment will be installed on drill rigs and radio communication will be used to deploy any stench gas or warning systems. A “Level 2” Ventilation on Demand (VOD) may be warranted for live monitoring of mine air temperature, flow, and chemistry to allow for semi-remote adjustments of ventilation controls. VOD studies are shown to pay themselves back under five (5) years of operation, however, to keep mine ventilation systems simple and cost effective no VOD systems have been incorporated to this study.

UG rock temperatures are expected to exceed 50 °C at depth. This coupled with additional heat loads of diesel and electric equipment, and the natural semiarid surface environment drives a requirement for mechanical refrigeration.

Heat loads from various sources are calculated through both manual heat balancing and through VentSim™ modelling. The total heat load on the mine ventilation system is estimated at ~13 MW. An external refrigeration of 7.5 MWR is required to maintain the workplace temperatures below a safe working temperature of 27.5°C wet bulb. The refrigeration is accomplished by a surface bulk air cooler (BAC) unit, which comprises of three water cooled chillers and two air cooled chillers.

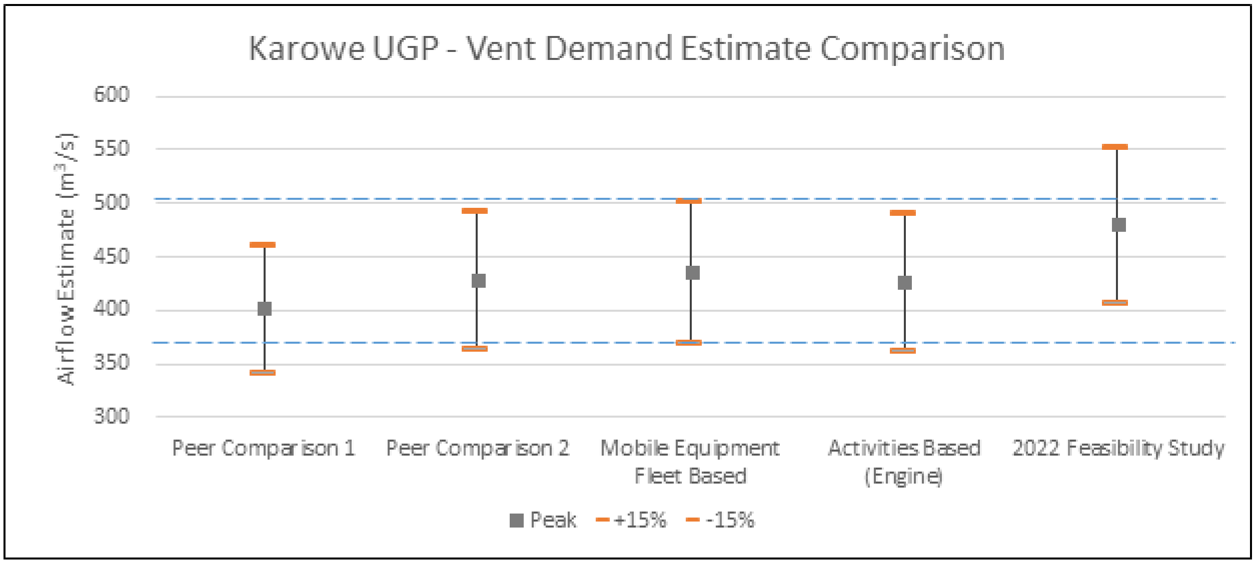
16.8.3.2 Ventilation Demand Estimate

The peak airflow demand for the KDM UGP is estimated through multiple approaches, which are:

- Approach 1: Peer comparison based on benchmark data available from other caving mines (JDS file data);
- Approach 2: Baseline comparison method of John Marks (2012);
- Approach 3: First principles – based on diesel equipment fleet (Top-down Method); and
- Approach 4: First principles – based on workplace activities / crews (Bottom-up Method).

The final estimate is based on Approach 3 (Diesel Fleet based), which indicates a peak requirement of 435 m^3/s. The estimated flow based of these approaches is summarized in the graph below (Figure 16-56). The surface main fans are sized based on the peak ventilation demand.

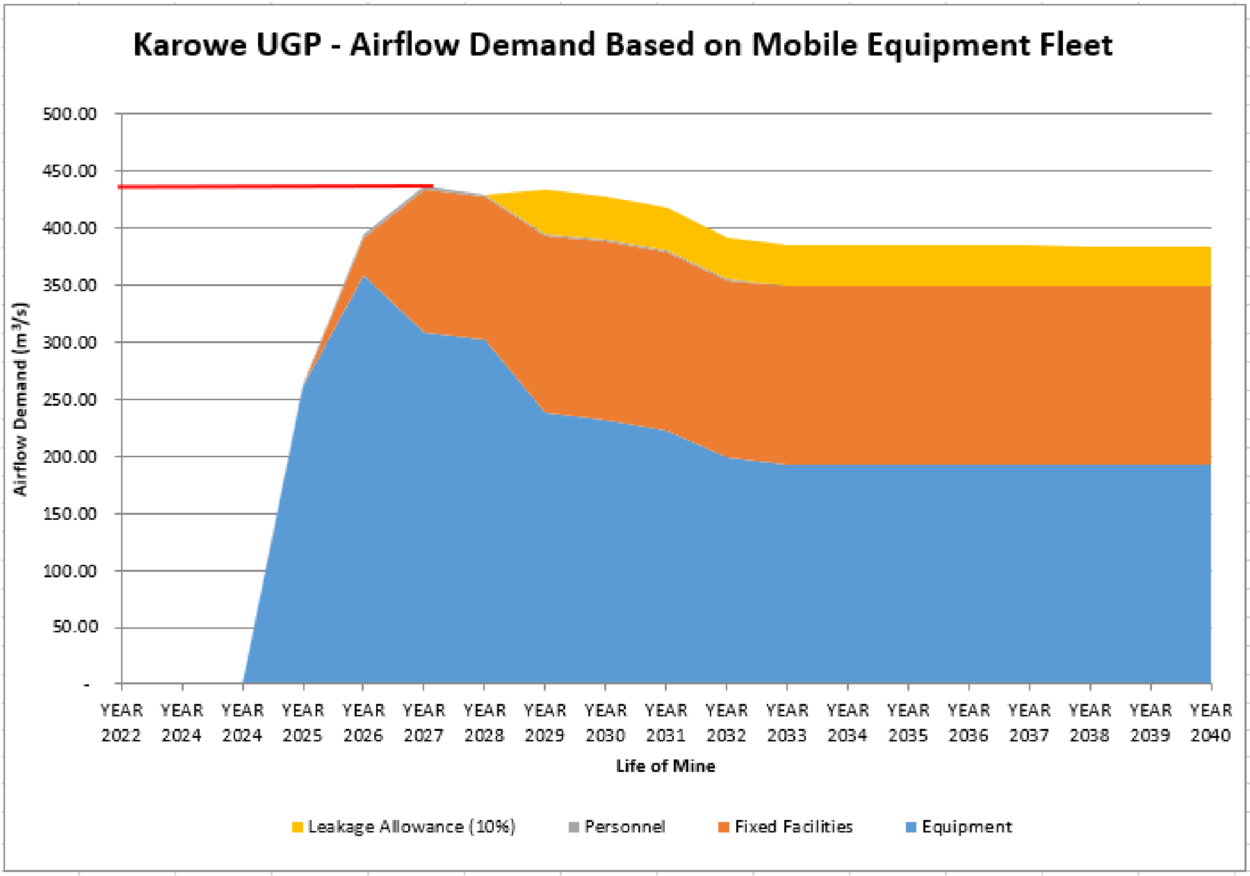
Figure 16-56: Ventilation Demand Estimate Comparison



Source: JDS (2023)

The LOM ventilation demand profile based on diesel equipment fleet over the life of mine is shown in Figure 16-57. The airflow requirement ramp-up shows a conventional stepped pattern associated with the overall stages of the mine development, production ramp-up, and steady state production. Per Botswana regulations, no utilization factor is applied for primary or secondary equipment, i.e., airflow demand is based on rated kW rather than utilized kW. The airflow requirement estimated in this approach is taken as the basis for designing the ventilation system and sizing the main fans.

Figure 16-57: LOM Ventilation Demand Profile – Equipment Fleet Based



Source: JDS (2023)

16.8.3.2.1 Main and Booster Fans

VentSim™ modelling was performed at various development and production stages to determine the sizes of the main and booster fans. During steady state production phase, a total of 463 m^3/s enters the mine through P/S. Of this value, 366 m^3/s is pulled by main RA Fans 1 and 2, and the remaining 105 m^3/s is handled by the twin booster fans on 670 L. After accounting for auto compression and system leakage, the resulting UG airflow achieves the target demand of 436 m^3/s. The VentSim™ model for this phase is illustrated in Figure 16-58.

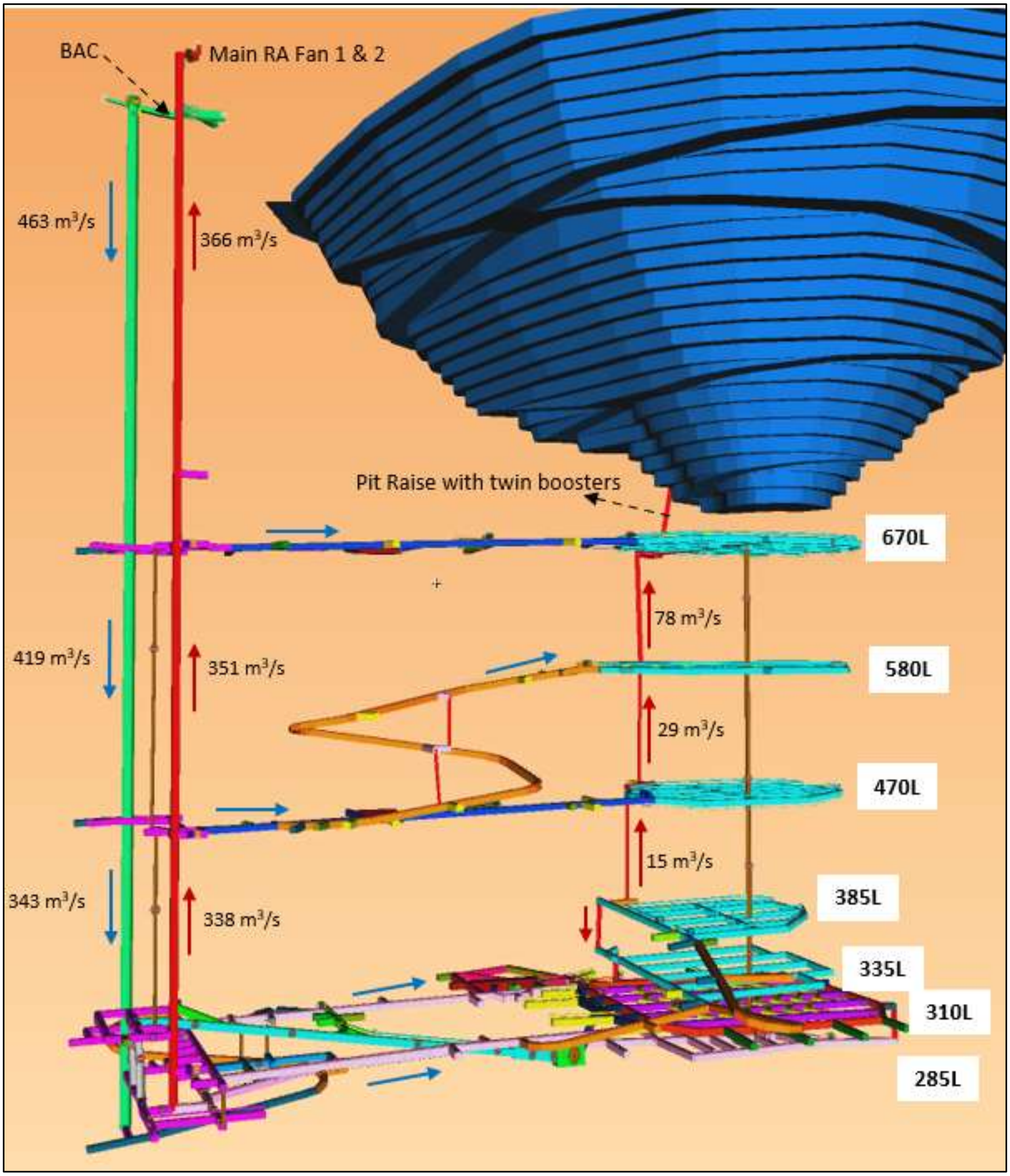
Both the main fans will be equipped with variable frequency drive (VFDs) to be able to operate at various operating points over the life of mine as necessary. The fans will also have temperature and vibration monitoring equipment, and PLCs to control the airflow from a local HMI. As shown in Figure 16-58, the shaft top bend will have enough room to accommodate an emergency escapeway bullet.

Fan motors will be externally mounted and connected by drive shaft to minimize corrosion through interaction with the saline water expected UG. Exhaust discharge will be vertical to minimize noise on the working terrace. Exhaust fans will be situated Southeast of the fresh air intake, against the prevailing wind direction, to minimize recirculation of exhaust air. Fans will be powered by dedicated transformers and MCCs, fed by a single 11 kV cable connected to both the main grid and back-up diesel generator banks in accordance with the Botswanan MQMA regulations (548-1).

UG booster fans will be installed in parallel arrangement in a bulkhead located on 670 L access drift to the pit raise. There will be mandoor airlock in between the fans to facilitate personnel access for maintenance purposes. Booster fans will be powered by a 525V mine power center (MPC) with shared load between other UG infrastructure and connected to the back-up diesel power supply akin to the surface fans.

One 75 kW emergency fan will be installed in a bulkhead immediately in front of the V/S on 670 L, 470 L, and 310 L shaft stations to facilitate emergency evacuation and to provide backup ventilation in the event of failure of the main fans.

Figure 16-58: Ventilation Schematic – Steady State Production Phase



Source: VentSim™ Model (2023)

16.8.3.3 Auxiliary Ventilation

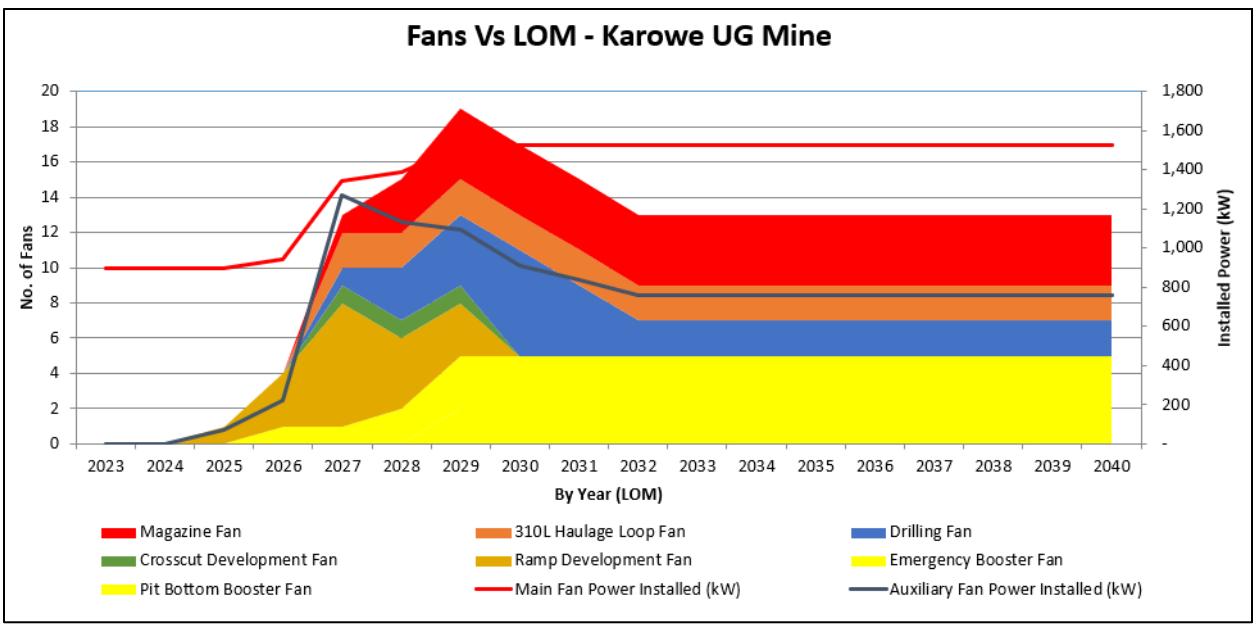
UG mining activities and fixed facilities will require auxiliary ventilation. The airflow requirement for each fan is based on the crew that fan needs to support, the size of the duct, length of the duct through which the airflow is pushed, and the duct leakage. The duct is sized to accommodate the duct hung on back of the wall with enough clearance between the largest equipment and the duct, and from the side walls. As development and production changes over time, so does the auxiliary ventilation needs. A total of 29 auxiliary fans are required with power ratings between 4kW to 75kW to meet the needs of the mine plan. As mine areas are decommissioned ventilation infrastructure will be relocated to new areas.

16.8.3.4 Total Ventilation Power Demand

The total peak installed ventilation power demand is estimated at 2.62 MW and occurs during the transition from development into production where peak lateral development occurs in parallel to the onset of large-scale drill and blast activities (Year 2027→2029). The respective peak electrical demand to operate the fans is approximately 1.97 MW.

Life of Mine ventilation power is illustrated in Figure 16-59. Assumptions include electrical demand for the main and booster fans at 80% of the installed power, and 70% for the auxiliary fans.

Figure 16-59: Fans and Electrical Load Profile Over LOM

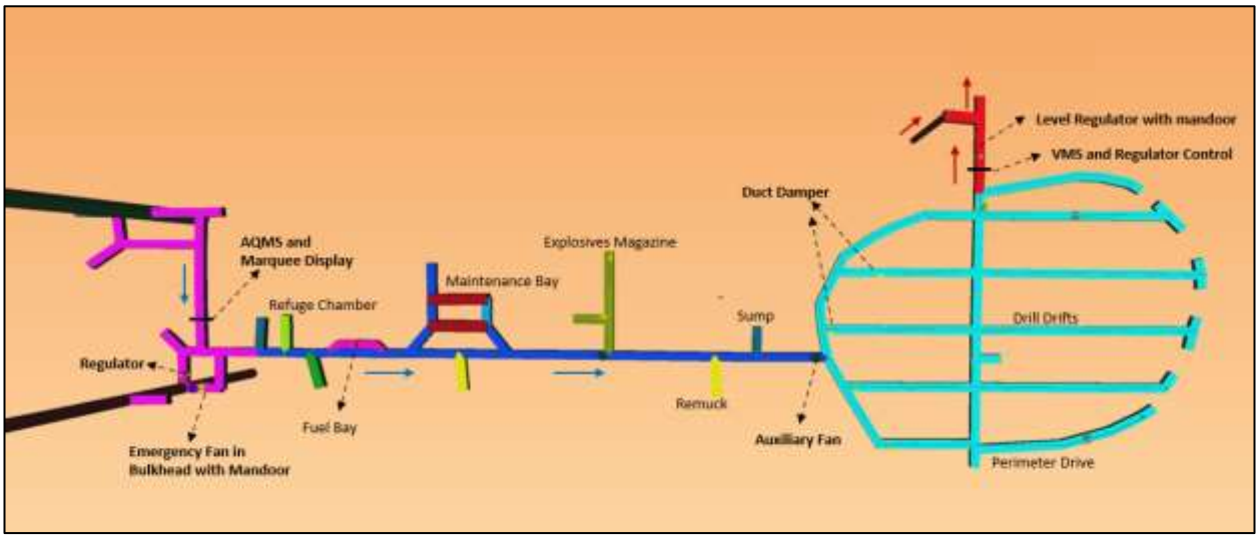


Source: JDS (2023)

16.8.3.5 Primary Ventilation Controls, Regulators, Doors, and Bulkheads

Each level of the KDM UGP has multiple ventilation controls and monitoring equipment. Figure 16-60 shows the arrangement of these infrastructure on a typical level at the KDM UGP.

Figure 16-60: Typical Level Ventilation Controls



Source: VentSim™ Model

The following are the various ventilation controls present in the KDM UGP ventilation system:

- <u>Mandoor</u>: There are 43 mandoors planned in the system over the life of mine. Some are standard mandoors in bulkheads, while some other are part of regulator and control door installations;

- <u>Control Door and Airlock</u>: There are three airlocks that provide a means to transition equipment and/or personnel from one portion of the ventilation system to another without disrupting the airflow balance between the circuits. These installations are sized to provide enough clearance for equipment to pass through. Additionally, there are control doors (equipment access doors) that are installed to provide a ventilation control that may be opened without significant disruption to the overall ventilation flow. In a single door application, the ventilation pressure across the door must be sufficiently low that the ventilation system will not be disrupted while the door is open and that the resulting airflow velocity through the door does not create a hazard to personnel or equipment;

- <u>Regulators</u>: A total of 18 regulators are planned for installation, at the return air access of each level and also at the rear end of each panel drive on 310 L. The louvers can be positioned to a desired setting to maintain a fixed airflow rate or to vary the airflow rate based on need. A mandoor will be located as required in regulator bulkeads to provide maintenance access where alternate access points are not available;

- Fire Door: At the KDM UGP, fire rated roll-up doors will be installed at the entrances to the UG shop as it contains combustible materials such as fuels, lubricants, tires, etc. that would pose a smoke, heat, or otherwise excess risk in the event of a fire. Similarly, they will be installed at the conveyor drive access for fire management; and

- Brattice: During shaft sinking process, development crews carry out lateral development of each shaft station. Once the level break-through is achieved, to avoid airflow leakages from one shaft into the other, brattices would be employed on a temporary basis until bulkhead construction is done.

16.8.3.6 Ventilation On Demand

Ventilation On Demand is recommended for future ventilation system designs, however, to maintain a simple and low-cost capital construction schedule a VOD system has not been included in project plans or budgets. All ventilation infrastructure will be manually operated using basic tools and or HMI interfaces at the local power source.

16.8.3.7 Emergency Alert System

In the KDM UGP, "stench gas" system will be provided to give an alert to the personnel UG who may be out of contact through other means. Typically, a mercaptan gas is used for a stench system due to the strong and distinctly disagreeable odour associated with sulphur in the gas. Stench gases are typically atomized into the main intakes (P/S in this case), and – where used – into the main air compressors so that the gas can be distributed throughout the mine. UG drill equipment will be equipped with gas sensors and radio communication between the operator and a supervision will be used to manually execute the emergency alert system.

16.8.4 Mine Air Cooling

The KDM UGP is located in the Orapa-Letlhakane region, the climate of which is semiarid to arid and is characterized by hot, wet summers and dry, dusty winters. The highest temperatures are experienced during summer with average temperatures ranging from 16.8°C to 35.9°C. During the winter months, the average minimum temperature is about 5.5°C and maximum is about 31.4°C. Geothermal gradients are greater than 4°C/100 m and strata temperatures are expected to exceed 50°C at shaft bottom.

At KDM, a working temperature of 27.5°C wet bulb requires heat stress management, 29.5°C wet bulb requires stop and correction action, and >32°C wet bulb requires removal from the environment.

It is anticipated that refrigeration of intake air is necessary to support both development and production during summer conditions. Manual heat balance estimates bolstered by VentSim™ modelling indicates that a peak refrigeration demand of approximately 7.5 MWR is required to maintain workplace temperatures below 28.5°C Wet Bulb Globe Temperature (WBGT) and/or level reject temperatures below 27.5°C wet bulb.

16.8.4.1 Design Criteria

The main design criteria used in designing the KDM UGP's refrigeration system include rock geophysical properties, ambient air conditions during summer, geothermal gradient parameters, and design reject temperatures (Table 16-13).

The peak refrigeration demand is estimated based on the worst case (design basis) surface temperature and thermal load from the mine.

Table 16-13: Refrigeration Design Criteria

Parameter	Value
Surface Datum Elevation above Sea Level	1,015 m
Surface Datum Barometric Pressure	89.7 kPa
Surface Datum Temperature Dry Bulb	29.0°C
Surface Datum Temperature Wet Bulb	22.5°C
Surface Datum Rock Temperature	21.5°C
Rock Density	2,693 kg/m^3
Rock Specific Heat	0.79 kJ/kg°C
Rock Thermal Conductivity	3.0 W/m°C
Rock Thermal Diffusivity	1.410 X 10^{-6} m^2/s
Rock Wetness Fraction	0.25
Airway Age	Based on Mine Schedule (or 5 Years)
Geothermal Gradient	4.6°C/ 100 m
Design Wet Bulb Temperature	27.5°C
Target Workplace Reject Wet Bulb Globe Temperature (WBGT)	28.5°C
Target Thermal Work Limit (TWL)	• 140 W/m^2 unrestricted work • 140 - 115 W/m^2 workplace precautions required • < 115 W/m^2 shift lengths need to be monitored

Source: Letlhakane Meteorological Service Station Data and Site Data

16.8.4.2 Heat Loads

16.8.4.2.1 Diesel Equipment

Diesel equipment heat is calculated based on the engine efficiency, diesel fuel consumption, and utilization factor. Based on the crew sizes and requirements, the estimated heat load generation from diesel equipment during steady state production (Year 2029) is 2.2 MW.

16.8.4.2.2 Electrical Equipment

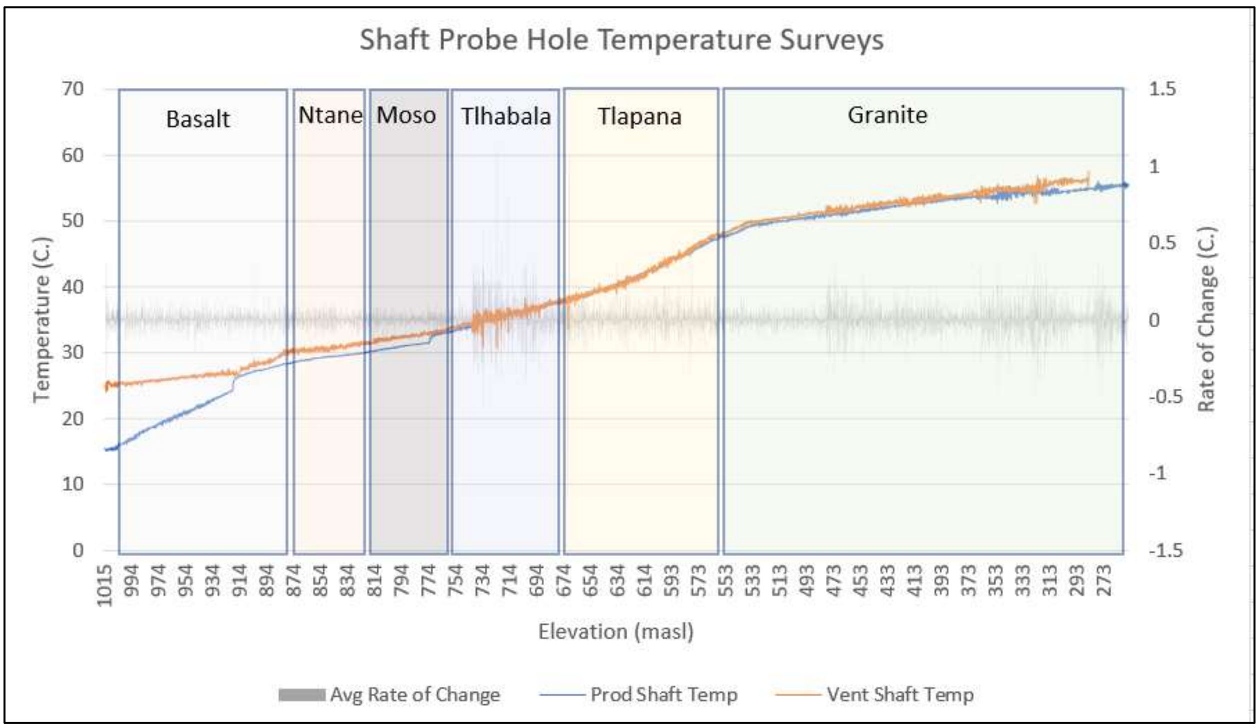
For electrically powered equipment, such as auxiliary fans, all electrical energy consumed is rejected as heat. This holds true for other electrically powered equipment as well, except for pumps when a portion of the consumed energy is used to move water. The total estimated heat load is 1.4 MW.

16.8.4.2.3 Geothermal Gradient

In late 2020 and into early 2021 two investigative core holes were drilled along the shaft centerlines of both P/S and V/S to be constructed as primary/secondary egress to the UG mine. As part of the investigation both core holes were probed for water temperature readings, which directly correlate to the geothermal gradient of the UG mine.

The temperature and water conductivity logs were measured using a Geotron fluid conductivity and temperature sonde. Results of these probing exercises are shown in Figure 16-61.

Figure 16-61: Geothermal Gradient



Source: Poseidon (2021)

It should be noted that there is a large discrepancy in the geothermal gradient between the two shafts near surface, with a variance of nearly 0.8 degrees per 100 m. Given the shafts are 100 m apart it is unusual to see this change unless environmental conditions change between surveys. It is also noted that there is a distinct difference in the surface temperatures for both holes which is likely influenced by the presence of stormwater in the shallow lithology units. The V/S was surveyed prior to the rainy season while the P/S was surveyed between periods of extreme storm and flooding events. As a precautionary measure the geothermal gradient was re-evaluated by looking solely at the temperature change from 900 masl where the two shaft water temperatures normalize. Table 16-14 outlines the results of this exercise and offers comparable results between shafts which are more in line with expectations.

Table 16-14: Geothermal Gradient Evaluation from 900 masl to Shaft Bottom

Shaft	Depth (m)	Max Change (C.)	Gradient (C./100 m)
Production	646.45	28.78	4.45
Ventilation	616.95	29.54	4.79
Average	**631.70**	**29.16**	**4.62**

Source: JDS (2021)

VentSim™ automatically calculates rock strata heat based on a user defined geothermal gradient. Peak heat load from rock strata is 3,500 kW for Year 2029.

Additionally, VentSim™ automatically calculates rock strata heat as it is broken through crushing and blasting. Heat from both development and production rock is input in the form of advance rate in VentSim™ and is estimated at 780 kW.

16.8.4.2.4 Auto-Compression

The phenomenon of auto compression is associated with the conversion of elevation (potential energy) to heat and pressure that occurs adiabatically. VentSim™ automatically calculates the auto-compression heat load, and for Year 2029, it is estimated at 3.2 MW. A manual calculation is carried out based on airflow distribution between leach level which estimates heat load at 3 MW.

16.8.4.2.5 Surface Heat Load

The design intake ambient air temperature is 22.5 °C wet / 29.0 °C dry. The long-term average wet-bulb temperatures exceed 22.5 °C only 3% of the time, which is reasonably close to 95% conditions – that is 95% of the year, the temperatures are below these values. For Year 2029, the natural cooling capacity of the intake air is estimated at 5.2 MW.

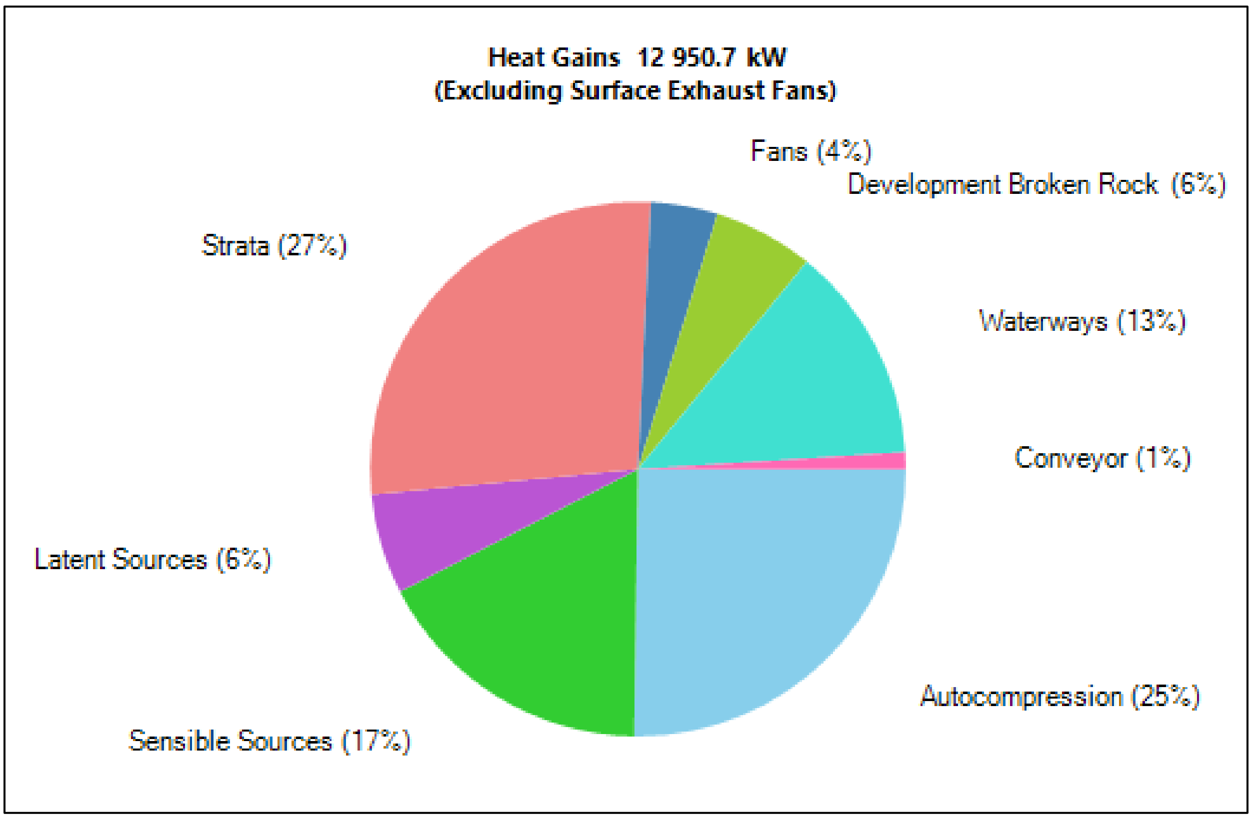
16.8.4.2.6 Ground Water

There are several known water strikes and permeable lithologies from which, the KDM UGP will experience high ground water inflows, from initial development through steady state production phase. Ground water is assumed to enter the mine at the virgin rock temperature (VRT) of the respective elevation and lose some temperature depending on the length of the path water must travel before entering the main sumps or pump station in the return airway. VentSim™ estimates the total heat load from ground water at 1.68 MW.

16.8.4.2.7 Thermal Balance

Thermal modelling of the KDM UGP during steady state production phase (Year 2029 in VentSim™) indicates the following heat loads (Figure 16-62). The total heat load on the system in steady state production conditions is 12.9 MW, excluding the surface main return fans (RA Fan 1 and 2), and twin UG booster fans. The natural cooling capacity of the intake air (5.2 MW) leaves a required mechanical refrigeration of 7.5 MWR.

Figure 16-62: Heat Load Distribution – VentSim™ Model



Source: VentSim™ Model (2023)

16.8.4.3 Refrigeration Plant

The proposed surface cooling plant (Figure 16-63) comprises of a bulk air cooler (BAC) with five chillers at a capacity of 1.46 MWR each, providing a peak combined refrigeration capacity of approximately 7.5 MWR.

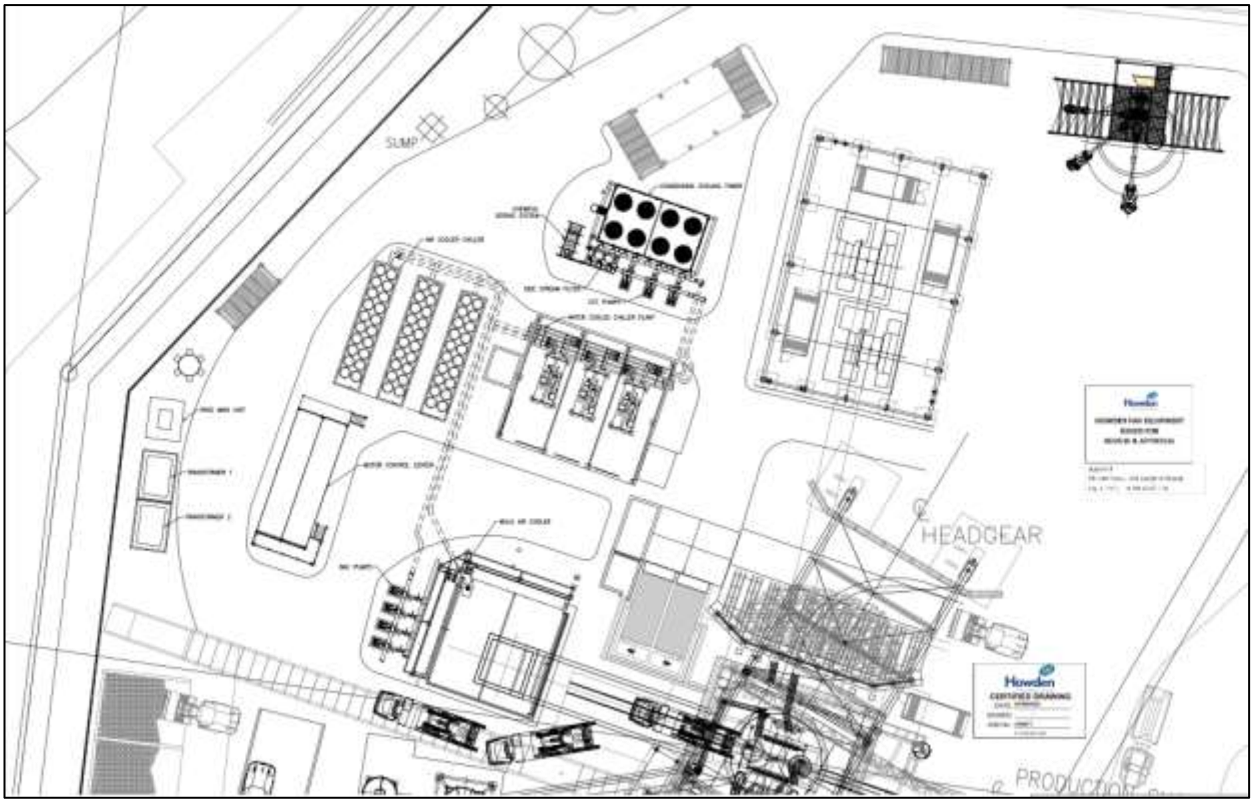
A subterranean concrete plenum of 4m x 4m dimension will serve as chilled air intake to the P/S. On top of this plenum will sit a large diameter forced air ventilation fan capable of pushing 163 m^3/s (178 kg/s) into the P/S barrel. Surrounding the plenum fan will be a fully enclosed bulk air cooler with large cooling coils mounted to the walls.

Outside of the BAC a series of chillers will cool reverse osmosis treated water to a minimum 8°C and pump this chilled water at a rate of 200 kg/s through the cooling coils. The plenum fan will draw ambient air at 29°C through the coils and cool the air to 10.8°C before entering the plenum and reporting to the shaft.

The chillers cool water through a heat exchanger, which also produces hot water. The hot water is pumped to a series of cooling towers which cool the air naturally by spraying into ambient air. A sump at the base of the cooling towers collects the sprayed water and pumps it back to the chillers for re-use. Water lost through evaporation in the cooling towers is made up by pumping reverse osmosis treated water to the cooling tower sump. At peak demand the make-up water requirement for the refrigeration system will be approximately 3 kg/s or 220 m^3/day over 20 operating hours and be supplied by the existing mine water treatment facility. This facility was expanded in 2021 to meet the additional demand of the BAC.

The surface exhaust fans mounted to the V/S will draw air down the P/S at 463 m3/s. The BAC plenum fan will push 163 m3/s of chilled air into the Shaft barrel and the remaining 300 m3/s will be drawn from the Production Collar at an ambient temperature of 29°C, mixing to provide an average intake temperature of 22°C.

Figure 16-63: Proposed Refrigeration Plant at P/S Collar



Source: LUCKARO5E_1161_MEC_DAL_001_1 (Howden 2023)

16.8.5 Water Supply

A combination of hot dipped galvanized carbon steel and HDPE piping will be used as required by specific water supply pipeline design or duty requirements.

16.8.5.1 Service Water

A 100 mm NB galvanized carbon steel pipe will be installed in the P/S which will connect to pressure reducing stations on each shaft station. An equivalent, 140 mm DN HDPE pipe has been selected for service water beyond the shaft stations, HDPE is more resistant to corrosion, cheaper, and faster to install, however it has a much lower pressure rating when compared to the carbon steel shaft piping. Service water pipe sizing is based on 340 m^3/day estimated maximum UG consumption rate and validated against a 1,200 m^3/day maximum consumption rate rule of thumb.

16.8.5.2 Potable Water

A 50 mm NB galvanized carbon steel pipe will be installed in the P/S which will connect to pressure reducing stations on each shaft station. An equivalent, 63 mm DN HDPE pipe has been selected to distribute potable water beyond the shaft stations. Potable water pipe sizing is based on a 220 m^3/day estimate maximum UG consumption rate.

16.8.5.3 Fire Water

A 150 mm NB galvanized carbon steel pipe will be installed in the P/S which will connect to pressure reducing stations on each shaft station. The 150 mm NB galvanized carbon steel pipe will be utilized to distribute fire water beyond that shaft stations, HDPE is not suitable due to the high-pressures of the fire water system. The Fire Water pipe sizing was performed by FireCo in 2021.

16.8.6 Electrical Distribution

The site is currently furnished with 11 kV power distribution which feeds various shaft sinking electrical components at surface. The UG shaft area is provided with a 11 kV feed from the main project substation to the shaft distribution switchgear.

From the main project substation, permanent feeder cables will be installed in in the P/S connecting the surface infrastructure to medium voltage switchgear at each mining level. Each level within the mine will have a connection to the UG feeders, forming a dual circuit UG electrical distribution system.

Horizontal feeds, from the medium voltage switchgear, will report to mine power centers (MPCs) / mini-subs which will provide the low voltage power supply to the UG equipment. MPCs will in general be standardized, rated far above typical requirements (2 MVA), skid mounted, and air cooled, to facilitate quick installations and redundancy on the project.

MPCs will be installed near key infrastructure, as well as to support the advance of development equipment. Multiple MPCs will be installed on each level as the location is limited by load requirements, cable costs for permanent infrastructure, and voltage drop on infrastructure and the development jumbo. The main pump room is the only area that is not currently planned to use a standard MPC, as the load requirement in the area will better suit a purpose-built transformer installation.

Rack based motor control centers or gully boxes will be constructed near the MPCs in substation cut-outs to provide power to the fixed infrastructure and mobile equipment. Similarly, PLC panels and other controls and instrumentation equipment will be supported by these substation cut-outs.

The voltage system used at the UGP is based on the South Africa UG mines, as outlined below in Table 16-15.

Table 16-15: UG Voltage Systems

Voltage Systems	Value
Medium Voltage (distribution)	11 kV / 3 ph / 50 Hz
Low Voltage (jumbo, pumps, motors, and fans)	525 V / 3 ph / 50 Hz
Small Power (welding, tools, and lighting)	400 V / 3 ph / 50 Hz or 230 V / 1 ph / 50 Hz
Control Voltage (instrumentation)	110 VAX 24 VDC

Source: JDS (2023)

The electrical reticulation system will support a maximum demand load calculated at approximately 16.4 MVA of apparent power demand in Q3 2027. Power factor correction work is ongoing to reduce the apparent power demand and Detailed Engineering of the UG electrical reticulation system is commencing in Q4 2023.

While an electric mine fleet is not planned at this point in time at the UGP, a provision has been made for an additional 2 MVa of capacity, to support potential upgrades during the operating period. 2 MVa will be able to support either a tethered or a BEV loading fleet, as well as any potential changes in charging protocols. Similarly, capacity for future expansion to the UG Main Pumping station has been considered in the power demand requirements.

16.8.7 Mine Communications

The KDM UGP communications system consists of:

1) A fibre backbone;

2) A dedicated wireless or private LTE (pLTE) network intended to support the use of teleremote equipment UG;

3) A standard communications system for regular communication between personnel; and

4) An UG control room constructed on surface.

16.8.7.1 Fibre Backbone

A fibre network connects the Lucara Main Route to the data room local to the UGP. A fibre network is currently installed on surface, connecting key infrastructure as well as the shafts. Fibre is planned to be installed in the P/S and fibre will be installed on each level, which will connect various infrastructure and provide a backbone for the controls and instrumentation, and mine communications systems.

16.8.7.2 Teleremote Communication System Requirements

The UGP will potentially utilize a fleet of up to 10 teleremotely operated pieces of equipment, including up to LHDs, and 5 longhole drills. In order to facilitate the installation and use of this system, an UG fibre network with wireless or pLTE communications will be installed in the operating areas.

The system must provide adequate signal strength for every client (i.e., piece of equipment) operating in the system simultaneously. The minimum required rate is 10 Mbps per piece of equipment. With an anticipated fleet size of 10, the minimum required rate is 100 Mbps. Some additional strength may be recommended to accommodate any irregularities in system use. Coverage areas must overlap, and signal strength (RSSI) must meet a minimum of -55 dBm in all areas. In areas where handovers occur, both access points must be above this minimum requirement. The system must also meet a signal-to-noise ratio of 25 dB at minimum, and a minimum netlink capacity of 20 Mbps. Only 50-100 ms of latency or jitter are permitted in the system to allow the teleremote system to function adequately.

Communications systems vendors as well as the mobile equipment vendors are engaged to provide the most fit-for-purpose solution for the KDM UGP. While a standard wireless access point solution is still being considered at this time, a pLTE solution is being investigated at KDM.

The pLTE utilizes a leaky feeder cable as an antenna to distribute signal throughout the teleremote area. This provides a better coverage area, as well as more predicable data rates and latency. Equipment data is secured locally and far less susceptible to attacks and intrusion. A pLTE solution, as well as a future upgrade to 5G, is more aligned with future industry requirements and the development roadmap for UG Mine Communications Systems, when compared to a traditional wi-fi solution. While the pLTE solution has been implemented successfully in-county and regionally, engaging the Botswana Communications Regulator Authority (BOCRA) for spectrum licenses would be required, and has not yet been pursued.

16.8.7.3 Standard Communications System

Mobile equipment operators, light vehicles, and supervisors will be equipped with hand-held radios to communicate with personnel on surface. Communication protocols will be used to ensure safe travels on the ramps and decline. A leaky feeder system will be installed along the main drives on each level to provide standard communication.

16.8.7.4 Control Room

An UG control room will be located on surface. The control room facility will include several services, including a SCADA system, teleremote operations, and CCTV monitoring.

16.8.8 Compressed Air

The compressed air system comprises of 11 x 1,370 cfm electric compressors providing a total 15,000 cfm at 145 psi. Peak compressed air demand occurs during shaft sink when both shafts are blowing over after a mucking cycle and the VSM muckers are utilized to assist.

All 11 electric compressors are linked to a common header on surface. The header feeds two 150 NB carbon steel air lines down the shafts, such that each shaft can receive the full system capacity if needed.

The surface and shaft compressed air reticulation will provide approximately 50 m^3 of air storage. No additional receiver tanks are installed or planned. Mobile mining equipment will be provisioned with booster compressors and/or tanks to provide additional air pressure or volume required for penetration and or flushing.

Compressors are stored outdoors under cover. Condensate traps are installed after the compressor bank, before vertical descent to the shafts, and at every station UG. Power is provided by two independent transformers.

16.8.9 Explosives and Detonator Storage

The KDM UGP mine plan will the following explosive products:

- Bulk Emulsion;
- Packaged Emulsion;
- Detonators;
- Blasting Accessories; and
- ANFO Explosives (Lateral development during shaft sinking).

A surface Bulk Emulsion facility is currently on site and operated by the explosive provider Eanex to support both OP and UG operations. Bulk pumpable Emulsion is stored in two (2) silos of 30,000 kg capacity for a total surface storage capacity of 60,000 kg. The emulsion used for OP operations is different from that used in UG development, and until OP operations are complete the UG emulsion will be stored in stand-alone 40,000 kg silos within the emulsion yard.

Bulk Emulsion will be transported from the emulsion plant to the UGP area in 3,000 kg approved totes supplied by the explosive provider. These totes will be loaded onto flat deck rail cars by a telehandler and transported UG via the Man and Material cage. UG, the totes will be collected and transported to one of three (3) magazines and stored until ready for use. A mobile emulsion charger will pump emulsion from the totes into the on-board emulsion container and transport to the face where the product is sensitized as it is pumped into the blast holes. Empty Emulsion totes will be transported back to surface for re use.

Sensitized explosives and blasting accessories will be stored on surface in two (2) existing magazines of 7,750 kg capacity, currently managed by the site explosives supplier. As the OP operations conclude, these magazines will be fully utilized by the UG operations.

Blasting accessories will be transported through the P/S shaft in approved explosive boxes. Blasting crews will pick up the estimated quantities of blasting accessories required for each shift and deliver those explosives directly to working faces and explosives-loading equipment UG.

Excess explosives and accessories will be returned to a secure UG explosives' magazine every shift. UG magazines will store excess detonators and blasting accessories.

Magazine capacities have been sized based on one week's consumption requirements and are listed in Table 16-16 through Table 16-18 below. Note that not all magazines will store these volumes at once. As the mine method works from bottom to top of the ore body, lower-level magazines will be decommissioned as upper-level magazines are brought online.

Table 16-16: Magazine Capacity 670 Station

Item	Type	Quantity (kg or Units)	Cases/Bags
Emulsion	Oxidizer	97,650 kg	32 Totes
ANFO	Explosive	9,000 kg	400 Bags
Packaged Emulsion	Explosive	4,300 units	125 Cases
Detonator	Explosive	4,300 units	125 Cases

Source: JDS (2023)

Table 16-17: Magazine Capacity 470 Station

Item	Type	Quantity (kg or Units)	Cases/Bags
Emulsion	Oxidizer	97,650 kg	32 Totes
ANFO	Explosive	10,500 kg	450 Bags
Packaged Emulsion	Explosive	5,100 units	150 Cases
Detonator	Explosive	5,100 units	150 Cases

Source: JDS (2023)

Table 16-18: Magazine Capacity 310 Station

Item	Type	Quantity (kg or Units)	Cases/Bags
Emulsion	Oxidizer	97,650 kg	32 Totes
ANFO	Explosive	32,000 kg	1, 280 Bags
Packaged Emulsion	Explosive	15,500 units	450 Cases
Detonator	Explosive	15,500 units	450 Cases

Source: JDS (2023)

UG Magazines shall maintain the following offsets:

- 100 m from shafts;
- 100 m from stopes; and
- 10 m between Type A and Type B explosive magazines.

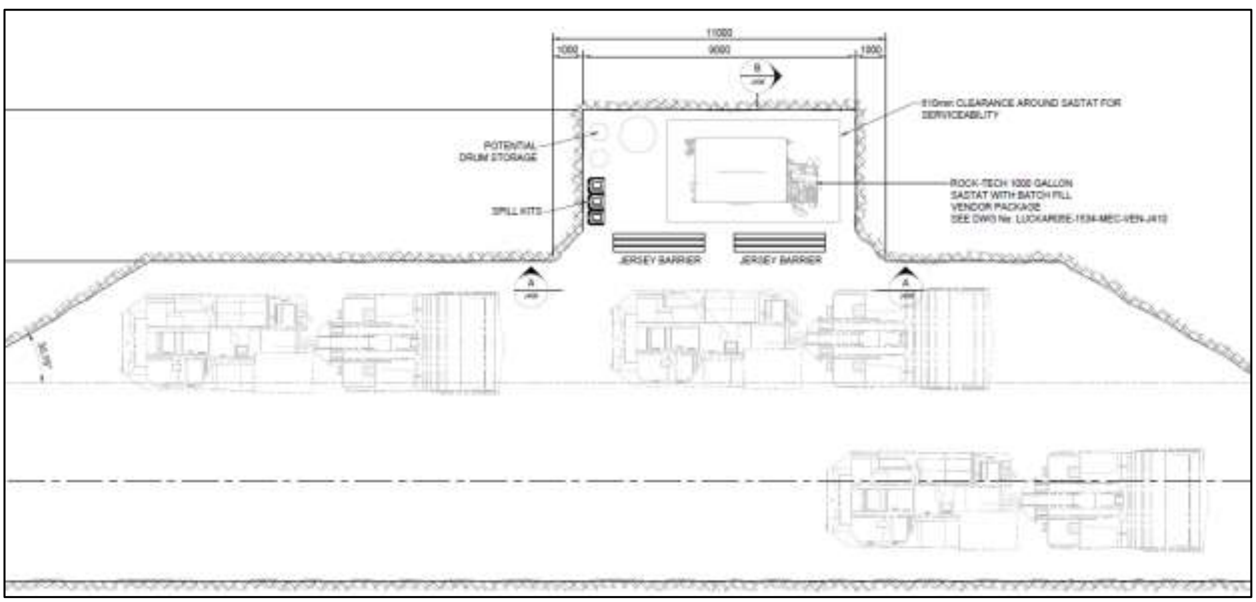
UG magazines shall not be restricted to return airways and may be constructed on primary causeways as required.

16.8.10 Fuel Storage and Distribution

An equipment fueling and lube station will be located near the shafts on 470 L and 670 L and will be able to provide fuel for the mobile UG equipment fleet. On the 310 L a larger fueling and lube station will be located near the drawpoints on 310 L to provide quick access for the production LHDs. Fuel will be transported UG daily in portable containers and pumped into the fuel dispensing equipment. No fuel lines will be installed in the shaft or by borehole. Pre-fabricated portable fuel dispensing equipment with built-in bund capacity, fire door, and suppression systems will be utilized to minimize construction efforts.

Figure 16-64 illustrates the type of fuel station that will be installed throughout the mine.

Figure 16-64: Fuel Bay General Arrangement



Source: JDS (2023)

16.8.11 Mobile Equipment Maintenance

The main UG maintenance facility will be constructed for services and repairs on 310 L. Small maintenance facilities will be constructed on the 470 L and 670 L to service minor repairs.

Main Facility access will be from the 310 L production drift and located in close proximity to the extraction area. The facility will be equipped with a wash bay, lube and oil change bays, electrical shop, tire storage, warehouse, and a general service bay with 15 t bridge crane.

The shop will be ventilated from 310 L production drive and will be connected to the exhaust drive for flow through ventilation. Fire doors will be installed to control ventilation during normal and emergency conditions.

The Workshop shall accommodate the peak mobile fleet that will be in use during the commercial production period. Five (5) maintenance bays will be required, as calculated based on a peak equipment fleet of 27 units, available work time, equipment operating forecasts, and service frequencies using the assumptions in Table 16-19.

A Maintenance Bay is a bay or allocation of space which is dedicated and furnished to perform equipment maintenance. One excavation or workspace may contain more than one Maintenance Bay if multiple pieces of equipment safely fit in the workspace.

Table 16-19: Workshop Sizing Assumptions

	Assumption	Quantity	Comment
Available Work Time	Work days per week	7	Monday - Sunday
	Maintenance shifts per day	1	Night Shift on call-out
	Shift Efficiency	75%	9/12 hr shifts
Major Services	Services per 2 weeks (minimum)	1	
	Equipment hours between services (maximum)	125	
	Major service duration (shifts)	1	
Minor Services	Services per 2 weeks (minimum)	2	
	Equipment Hours Between Service (maximum)	24	
	Service Duration (Shifts)	0.5	
Unplanned Maintenance	Unplanned repairs (ratio)	40%	
	Unplanned service duration (shifts)	5	
Mechanical Availability	Loaders Target M.A.	80%	
	Drills Target M.A.	80%	
	Other Target M.A.	80%	

Source: JDS (2023)

The workshop Flow must be continuous, safe and process oriented. The Maintenance Area will be one-way traffic and will have a single entrance and exit. Equipment will come from the extraction zone to the parking area should there be no space in the Workshop. Once the equipment is ready to enter the maintenance area, it will first report to the Wash Bay for a hot water pressure wash. Depending on the type of service required, it will either report to an Inspection Bay for lube service or to the Repair Bay for maintenance. Once the maintenance is complete, the equipment will exit the Workshop, report to the fuel and lube bay for top-ups, and proceed to the working face.

The Repair Bay is designed with the following considerations:

- Wide enough to accommodate a passing lane;
- Long enough to accommodate 2-3 machines;
- Tall enough to service the tallest machine at full reach;
- Equipped with one 15 t overhead crane and service cat ladder and platform;
- Include a system for dispensing and collecting lubricants and fluids, equipped with hoses and cable reels;
- Contain a catchment sump; and
- Accommodate sufficient space for storage and shop tooling.

The Inspection and Service Bay has been designed with the following considerations:

- Accommodate 1 to 2 pieces of equipment;
- Constructed with Inspection ramp and man-access trench for performing under vehicle work;
- Include a system for dispensing and collecting lubricants and fluids, equipped with hoses and cable reels; and
- Accommodate sufficient space for storage and shop tooling.

All hose and cable reels shall be mounted on a pivoting bracket and equipped with 10 m hoses or cables. The system for collecting lubricants and fluids in the pit shall consist of Quick connections/disconnections and pneumatic pumps for pumping the waste oil and, if needed, the environmental waste fluids to different tanks.

The Lube Store shall be within the Maintenance Area and will store and distribute lubricants to working bays. Dedicated access will be provisioned for handling equipment to deliver and remove lubricant drums and totes without entering the workshop. The Lube Store shall be positioned at the exhaust-side of the workshop to eliminate risk of fumes entering the working area. Fire doors with side-entry man doors shall be equipped at each entrance and exit.

The Lube Store shall be based on Rock Tech's pre-engineered solutions, minimizing the requirement for additional engineering and stick-building UG. The basic design concept includes a series of two (2) stacked containment tank oil dispensing units. The top tank is placed by a fork loader, which free-drains into the lower tank. The lower tank is permanently connected to distribution pumps which feed hose reels within the workshop. Once empty, the top tank is removed by fork loader and brought to surface for refilling. The system is designed to work with 1.0 m square totes commonly supplied by lubricant vendors. Unlike the fuel skids, these lubricant systems do not include pre-engineered bunds, fire doors, or fire suppression systems and will be constructed as part of the Lube Store works.

An Environmental Station shall be constructed near the Lube Store to temporarily store waste from heavy machinery repairs. The station will allow for bunding of waste oils and the loading and unloading of bins to be conveyed to surface for emptying. A well-organized environmental station improves handling and reduces risks by centralizing the waste storage for the facility.

During initial UG development all contractors will be responsible for mobile equipment maintenance and will be required to provide their own maintenance solutions until the main facilities are commissioned. These are typically prepared in old remucks and dual purpose with other infrastructure such as sump wash bays.

16.8.12 Mine Safety

A permanent refuge station will be located on the 310 L and will also serve as a permanent lunchroom. Self-contained portable refuge stations will be located on the 285 L, 340 L, 470 L, 580 L and 670 L. The refuge chambers are designed to be equipped with dedicated fresh air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers doors are sealed to prevent the entry of gases.

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the UG electrical installations, pump stations, fueling stations, and other strategic areas. Every vehicle will carry at least one fire extinguisher of adequate size. All UG heavy equipment will be equipped with automatic fire suppression systems.

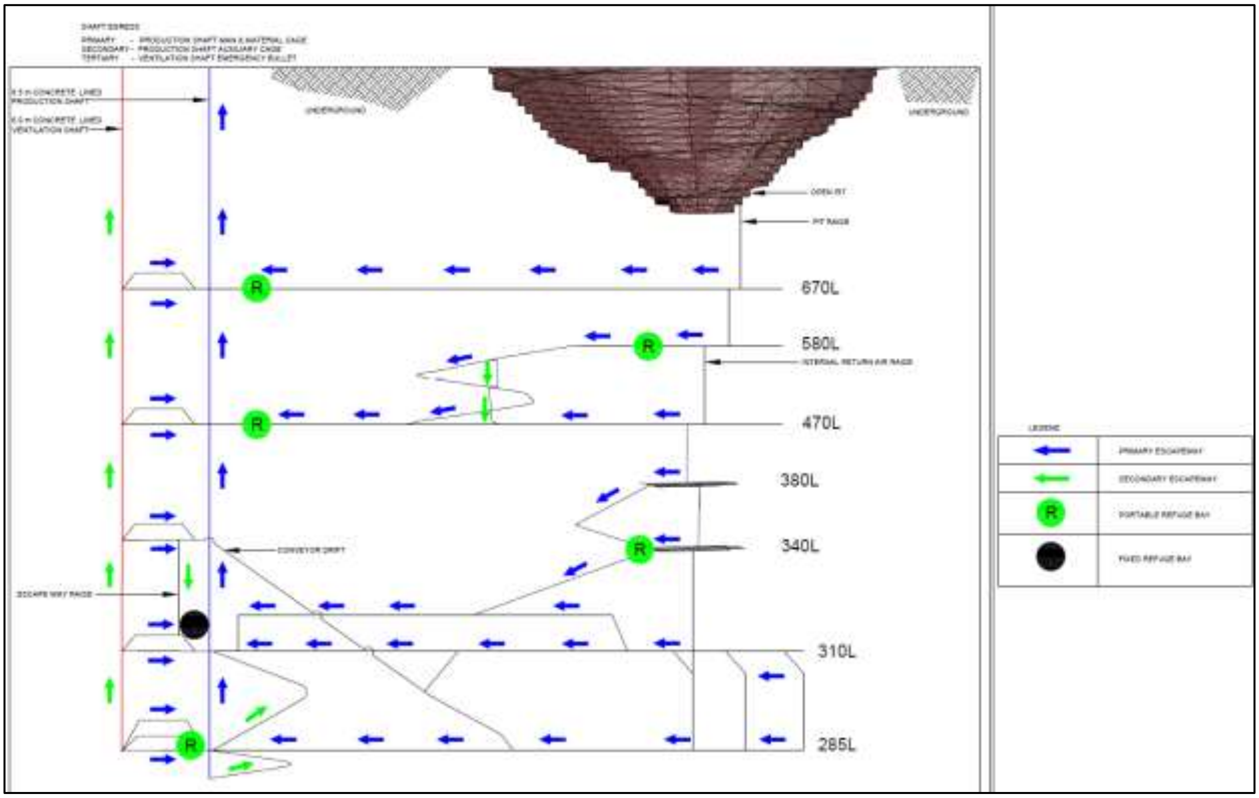
A fully equipped mine rescue team will be available every shift to respond to emergencies.

A stench gas system will be installed on the ventilation system and would be triggered to alert UG personnel in the event of an emergency.

16.8.13 Mine Egress

No ramp or ladderway is planned to connect the UG mine to surface. Primary mine access will be through the P/S via either the Man and Material Cage or auxiliary cage. Secondary emergency egress will be via the V/S mobile emergency mobile bullet winder. Figure 16-65 illustrates the location of primary / secondary egress routes along with planned locations of refuge chambers.

Figure 16-65: Mine Egress General Arrangement



Source: LUCKAR05E-1500-MIN-PFD-J119 Escapeway Long Section, JDS (2023)

Simulations were prepared by JDS to evaluate emergency egress design requirements. Absolute worst-case events that have the potential to render the primary conveyances unavailable and call for the use of an auxiliary egress system were evaluated.

During the initial phase of the evaluation, a list of plausible emergency situations was generated. This list included everything from equipment fires to individual medical emergencies. As the evaluation process was progressed, only those scenarios which could not be managed by the primary egress system were kept for further analysis. These scenarios include:

1) Mechanical failure – a severe incident in the shaft has the potential to damage vital shaft components and electrical infrastructure to the point that the man/material cage, service cage, and skips are inoperable, and power is no longer being provided to the UG mine;

2) Power failure – a prolonged outage in the National electrical grid system combined with a failure in the diesel power generation system can cause a situation in which power is no longer being provided to the UG mine; and

3) Fall of ground –a major fall of ground in the shaft has the potential to damage vital shaft components and electrical infrastructure to the point that the man/material cage, service cage, and skips are inoperable, and power is no longer being provided to the UG mine.

The effects of these events could lead to trapping of workers UG without a means of egress from the mine, which could further lead to exposure of several life threating conditions: temperature rise, flooding, and loss of ventilation. It was found that of all events, the bottleneck was ultimately the capacity of refuge chambers. In the Southern African region, it is uncommon and difficult to acquire a portable refuge chamber with back-up battery power greater than 48 hours.

This information led to the design of a mobile bullet winder which could fully evacuate the mine in less than 48 hours. Based on an expected peak workforce of 156, it was found that a 6-pax bullet would be required with a minimum hoist speed of 1 m/s and could theoretically evacuate the workforce in 32 hours.

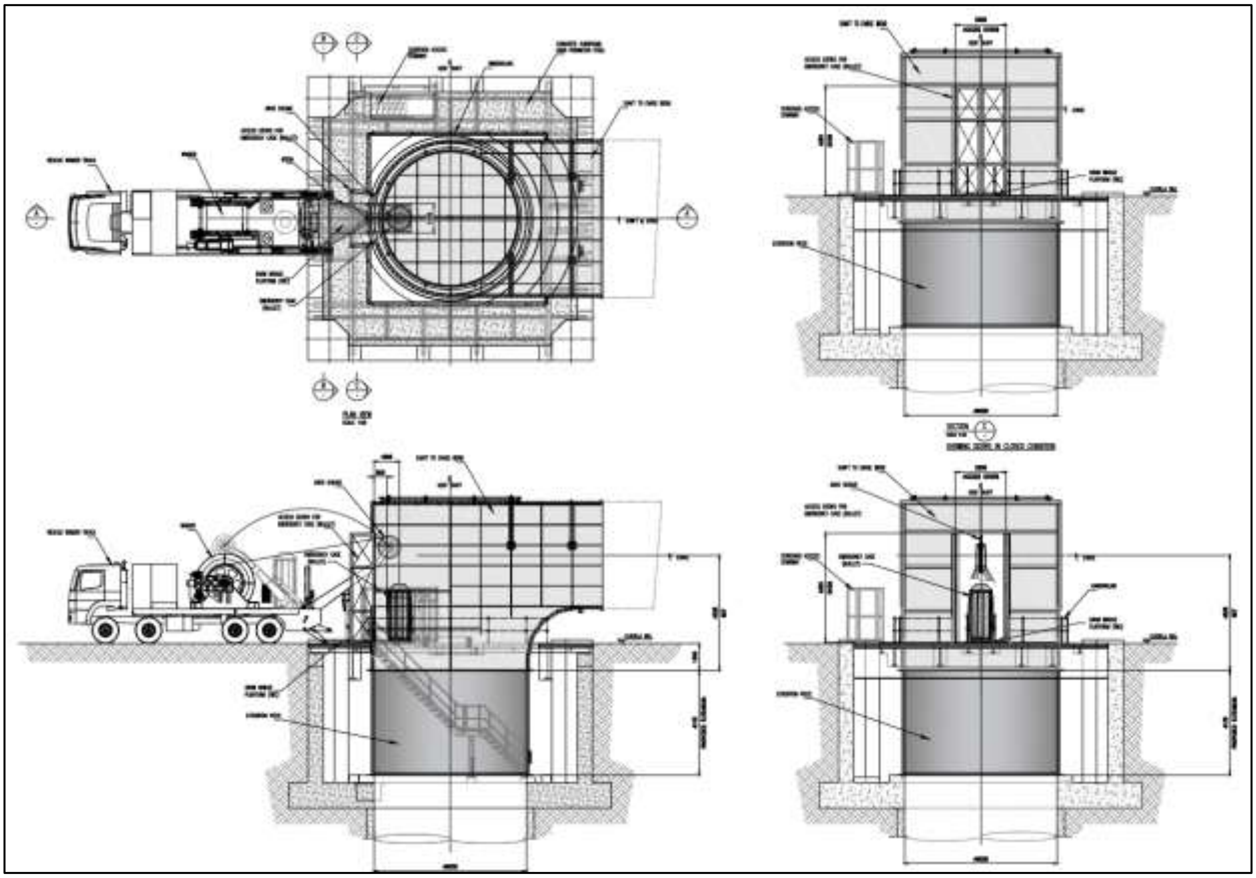
Table 16-20: Mobile Winder Evacuation Timing

Assumption	Men Working	Evacuation Timing (hrs)
310 Level Evacuation	110	17.3
470 Level Evacuation	26	4.1
670 Level Evacuation	20	2.8
Sub Total	**156**	**25.8**
Operating Efficiency		80%
Grand Total Evacuation Duration	**156**	**32.2**

Source: JDS (2023)

Figure 16-66 illustrates the configuration of the mobile bullet winder when deployed at the Vent Shaft collar. In this event the winder backs up to the shaft and deploys the bullet through the Ventilation Evase.

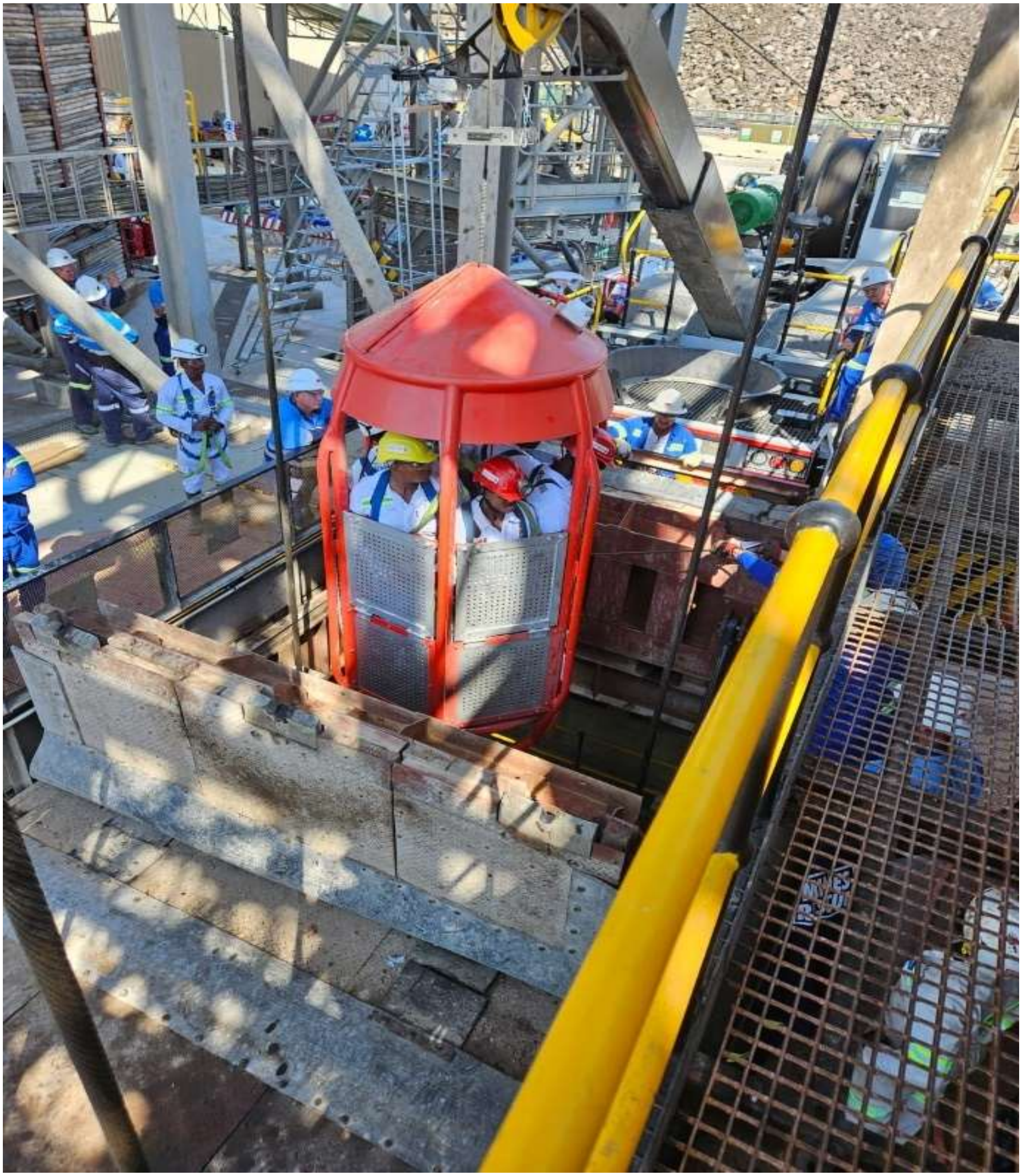
Figure 16-66: Mobile Emergency Winder Deployment



Source: LUCKAR05E-1270-S-DAL-V0818, V/S - Shaft Infrastructure Shaft Collar Permanent Collar Evase Interface, UMS (2022)

In 2023 the mobile emergency winder was commissioned and trialed on the P/S. This winder will also serve as emergency egress during the shaft sink as shown in Figure 16-67.

Figure 16-67: Mobile Emergency Winder Commissioning 2023



Source: JDS 2023

16.9 Unit Operations

16.9.1 Drilling

Drilling activities will be undertaken by the following equipment:

- Two boom jumbos; and
- In the hole hammer (ITH) longhole drill.

Drilling productivities (metre/percussion hour) were built up from first principles by drilling machine type and heading dimensions. Jumbo drilling rates average 75 m/hr in a 5.0 m x 5.0 m heading, and longhole drill machines average 12 m/hr or 105 m per shift.

16.9.1.1 Development Drilling

Development headings will be developed by two-boom electric jumbo drills. Jumbos will be equipped with 4.88 m (16') drill steel and will advance 4.4 m per blast. Jumbo advance is budgeted to an average of 3.5 m/d per machine in priority headings and 2.75 m/d per machine in non-priority headings, to a maximum 11 m/d per machine over four active faces. This equates to approximately 2.25 rounds per day per machine when four faces are available.

16.9.1.2 Production Drilling

Longhole production drilling will start with 45 m downholes drilled from the 380 level to the top of the drawbells. 165 mm diameter holes drilled on a 4.35 m burden and 5.00 m spacing will yield an average powder factor of 0.6 kg/t. This relatively short sub level with relatively high powder factor has been designed specifically to ensure high drill accuracy and high blast fragmentation to initiate the shrinkage operation.

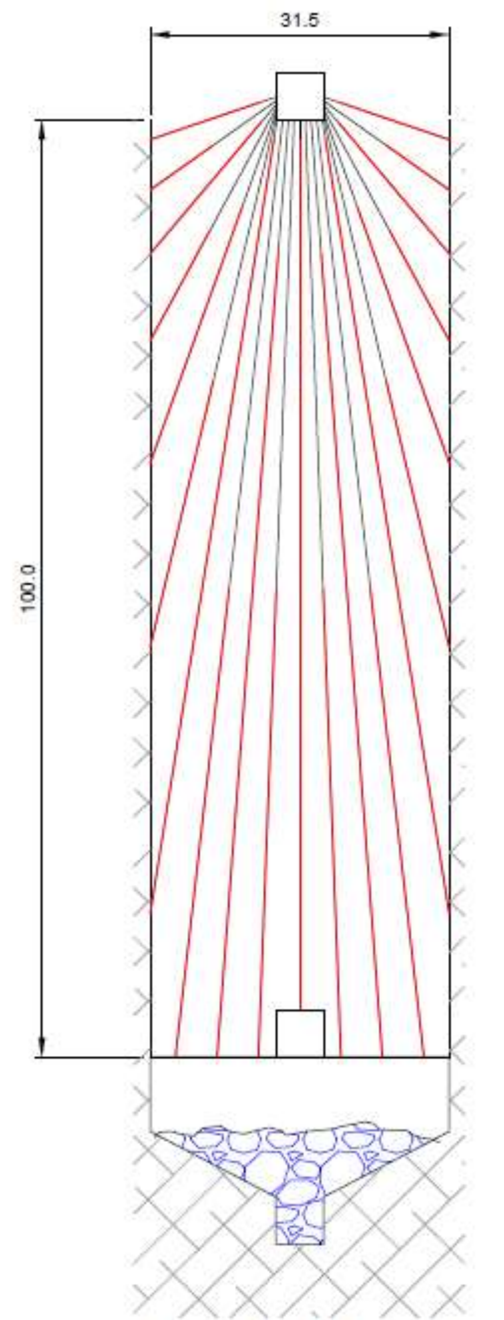
Above the 380 L, sublevels are increased to 100 m vertical spacing. Longhole drilling of mainly down holes with 150 mm diameter is planned on a 4.35 m burden and 5.00 m spacing to yield an average powder factor of 0.4 kg/t. This material will experience more comminution within the pipe as muck is pulled from the drawbells, so a lower powder factor will be used. The OP operations currently drill and blast ore to a powder factor of approximately 0.4 kg/t.

Some stoping would include drilling of upholes, particularly in the crown pillar.

The average drill length for a typical 100 m tall ring pattern is 58 m and yields 33.9 t/m drilled including a 10% redrill factor. Figure 16-68 depicts a typical ring design.

Figure 16-68: Long Hole Stope Ring Design

Drilling 380+	Units	Drilled (m)	Total Loaded (m)
Hole 1	m	14	12.26
Hole 2	m	16.31	8.15
Hole 3	m	20.7	18.24
Hole 4	m	28.01	13.24
Hole 5	m	39.67	28.85
Hole 6	m	58.16	29.08
Hole 7	m	85.7	83.68
Hole 8	m	100.63	50.32
Hole 9	m	100.28	85.99
Hole 10	m	100.07	50
Hole 11	m	100	98
Hole 12	m	100.07	50.00
Hole 13	m	100.28	85.99
Hole 14	m	100.63	50.32
Hole 15	m	85.7	83.68
Hole 16	m	58.16	29.08
Hole 17	m	39.67	28.85
Hole 18	m	28.01	13.24
Hole 19	m	20.7	18.24
Hole 20	m	16.31	8.15
Hole 21	m	14	12.26
Average Hole Length		58 m	
Drill Hole Diameter	mm		150
Burden (19 x dia)	m		4.35
Spacing (1.25 x B)	m		5.00
Stope Dimensions:			
Stope Height	m		100
Stope Width	m		32
Ore Density	t/m3		2.9
Blasted Tonnes	t		45,698
Drilled Holes per Ring	#		21.00
Drilled Metres per Ring	m		1,227
Redrill	%		0.10
Redrill Metres per Ring	m		123
Tonnes per metre drilled	t/m		33.9
Explosives	g/m3		1,200
Column Load	kg/m		21
Total Load	kg		18,186
Powder Factor	kg/t		0.40



Source: JDS (2019)

16.9.2 Blasting

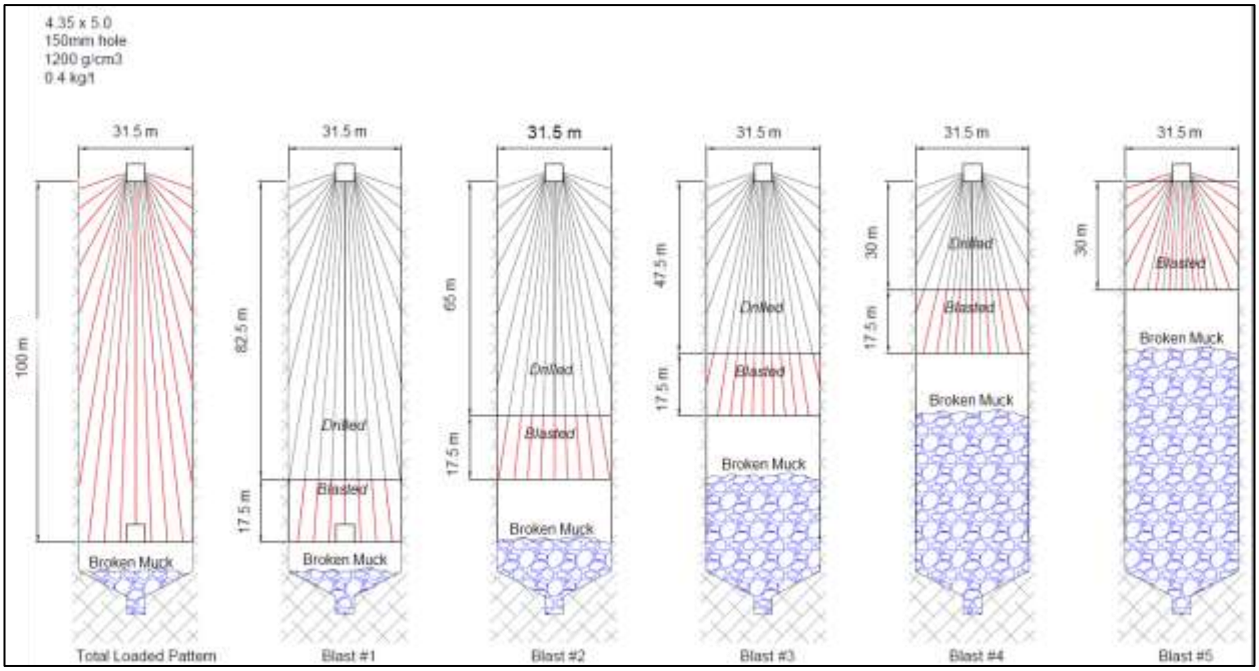
For explosives use, blasting crews will be trained and certified. Bulk emulsion will be used for production blasting and development rounds. Boosters, primers, detonators, detonation cord and other ancillary blasting supplies will also be utilized. Smooth blasting techniques may be used as required in headings, with the use of trim powder for loading the perimeter holes.

Bulk explosives will be manufactured on surface in accordance with current Botswana Explosives Regulations. The blasting crews will pick up the estimated quantities of explosives required for each shift using explosives cartridges and transport vehicles and deliver those explosives to working faces and explosives-loading equipment UG. Excess explosives and accessories will be returned to the secure powder magazine every shift. All explosives and detonators in and out of the magazines will be documented as per Botswana Explosives Regulations.

During the pre-production period, blasting in the development headings will be done at any time during the shift when the face is loaded and ready to blast provided all personnel UG are in a designated Safe Work Area and ventilation is adequate. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of each shift. Where ventilation allows, multi-blasting of isolated high priority development headings is possible.

Each 100 m tall stope will be blasted in several vertical segments, maintaining a minimum 30 m sill pillar below the drill panel until the final blast is taken and access to the drill panel is lost. Figure 16-69 illustrates the drill and blast sequence of a single stope.

Figure 16-69: Stope Blast Sequence



Source: JDS (2019)

Stopes will be blasted such that a dome shape is created across the South Lobe. This is to promote geotechnical stability within the lobe and prevent slabbing of large blocks into the muck pile.

16.9.3 Ground Support

Ground support will vary depending on the size of opening, service life, and ground conditions. Table 16-21 outlines the different ground support applications planned for KDM UG.

Table 16-21: Ground Support Regime

Support	Description
Temporary Support (ore)	Bolt and Welled Mesh 2.4 m backs and 1.8 m walls down to 1.5 m grade line above the floor 1.2 by 1.2 pattern (split set)
Permanent Support (waste)	Bolt and Welled Mesh 2.4 m backs and 1.8 m walls down to 1.5 m grade line above the floor 1.2 by 1.2 pattern (rebar)
Shotcrete	50 mm to be applied to all intersections and large excavations

Support	Description
Cable Bolting	At all intersections, 6.0 m cables to be installed on a 1.8 m x 1.8 m pattern
Drawpoints Additional Support	D-Bolts and mesh as primary support with cable bolts and shotcrete as secondary support
	Nose pillars to receive steel plate 1.5 m from the ground wrapped around nose of the herringbone pillar, post bolted with 6 m cables (twin-strand)

Source: JDS (2023)

Ground support will be installed in accordance with specifications based on geotechnical analysis for the various rock qualities expected. The massive (unstructured) nature of the of the kimberlite and granite renders the ground support design inapplicable to empirical systems such as RMR, Mathew's Q or modified Q. These systems rely on block size, jointing, water flow and joint condition, which are not applicable to unjointed rock masses. The ground support design has, therefore, been based on industry standards for life of the opening and function of the excavation. The proposed ground support has been evaluated by Itasca using Flac 3D to confirm suitability of the design during the various phases of the mine life. The proposed ground support was deemed suitable with the pyramidal opening sequence.

Primary ground support will be installed post-mucking of the blasted drift. No additional development will be commenced in the heading prior to the installation of primary ground support. At no time will mine workers be under unsupported ground. Secondary and tertiary support may be installed out of the development cycle by the service crew in accordance with the ground support management plan.

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with different stratigraphic units within the waste rock. Discretion will be made by the development lead as to which ground support is required, with additional review and recommendations provided by the on-site geotechnical engineer.

Electric-hydraulic jumbos, bolters, and shotcrete spraying machines will be used. Shotcrete will be applied when required as a wet mix, which is mixed in a transmixer and pumped into a skid mounted shotcrete sprayer.

Regular pull tests will be conducted on-site to ensure adequate installation of resin rebar, split set, and cables bolts are being done. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy in accordance with the ground support management plan and QA/QC.

16.9.4 Mucking

The LHD selected for development mucking has a 17 t (7 m^3) nominal capacity. For development, LHD's will typically muck a blasted round to a nearby re-muck bay in order to clear the working face prior to ground support installation. Rock temporarily stored in the re-muck is then either trammed to a rock pass or loaded into a haul truck.

There will be approximately 50 drawpoints over five extraction drives in operation throughout the life of mine. Material will be systematically mucked from the drawpoints by three LHDs to maintain the desired muck pile shape within the lobe. During drill and blast operations this shape will be a cone to mimic the dome shape created by the blast sequence. During final draw down the muck pile shape will be an inverted cone to maximize wall support until the lobe has been emptied.

Stope ore will be mucked with a 21 t (11 m^3) LHD and trammed directly to the crusher coarse ore bin grizzly. In the event the crusher cannot accept ore feed, either for capacity or maintenance reasons, the LHD will muck into one of several remuck bays located adjacent to the grizzly and later rehandled when space becomes available.

LHD cycle times and quantity requirements were calculated from first principals. An average haul distance of 150 m was used for the tram distance from the drawpoints to the grizzly.

Three production LHDs will be required to meet the target production rate. This has been calculated based on number of loads, cycle times, and available working hours per day. An Arena simulation was prepared to test the impact of LHD requirements during events of unscheduled maintenance and longer than average tram distances during periods of drawpoint rehabilitation. This simulation also concluded that three production LHDs would be required to meet production. Development LHDs will be available on standby to assist with production mucking if required.

LHDs will be inspected before each shift and returned to the maintenance facility at end of shift for fueling, lubrication, and preventative maintenance (PM) if required. LHDs are expected to require refueling every seven operating hours and will report to the fuel station some 200 m from the working area.

Diesel fired LHDs have been selected for all mucking activities at KDM.

16.9.5 Crushing and Conveyance

The Crusher and Conveyor System centers around a Telsmith 1,270 mm x 1,524 mm (50” x 60”) – 6 Piece Single Toggle Jaw Crusher. Production LHDs will transfer material to a static grizzly, above an ore pass, which feeds the jaw crusher. The jaw crusher product will be transferred by a series of three conveyors to one of two fine ore bins. Material from the fine ore bins will be transferred by a loading pocket conveyor, to 21 t skips which will be hoisted to surface.

The crushing and conveyance system has been described in detail in Section 16.7.10.

16.9.6 Hoisting

The loading pocket bins feed a skip loading conveyor, where material is dropped into one of two 21 t loading flasks which in term feed the 21 t bottom dump skips. Skips will be hoisted opposing to one another (when one is going up, the other is going down) on two-minute skip hoisting cycles. The average electrical power load for the rock hoisting cycle is 3,570 kW (RMS). The rock hoisting capacity is 3.2 to 3.5 Mt/a based on an annual average availability/utilization of 65 to 70%.

On surface the skips will dump into an elevated bin equipped with a dual "pant-leg" chute which reports primarily a conveyor and secondarily a truck loading chute when required. A short conveyor will report hoisted material to a surface stockpile for rehandle into trucks. When the conveyor is in need of service, or the mine wishes to change material feed (waste vs ore) the truck chutes may be employed to report material directly to 40 t trucks.

Ore will be trucked to the existing processing plant and waste trucked to the WRSF, both some 2 km away from the shaft.

16.10 Mine Personnel

Mine development contractors will be utilized for mine construction and pre-production operations. The mine plan envisions, for budgetary purposes, five separate mine development contractors; one each for

- Shaft sinking;
- UG development;
- Infrastructure installations;
- Raise boring; and
- Production drill and blast.

Several existing OP contract services will continue to support UG operations, including load and haul, and site road maintenance.

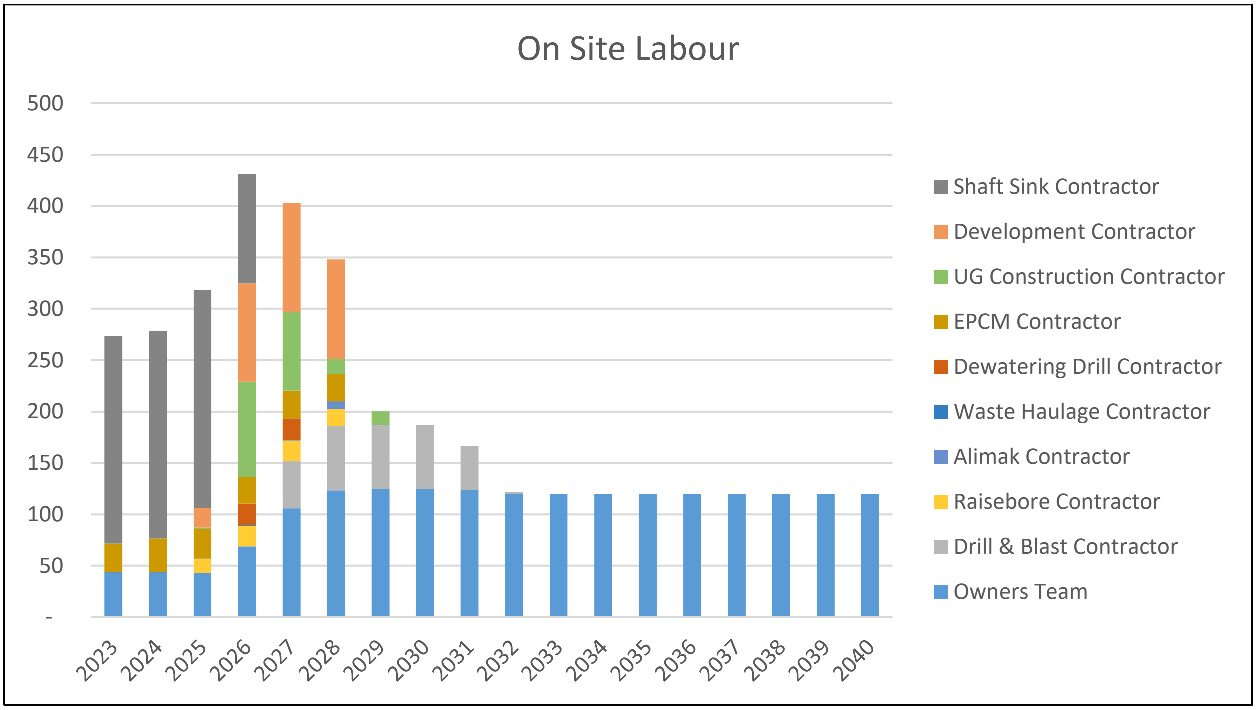
Existing OP employees will be trained and transitioned to the UG mine where possible.

All UG mine labour will operate on two 12-hour shifts, seven days per week. During mine construction contract labour will work a 2x1 schedule. During mine operations UG labour will work 2x2 schedule, equal to the current plant operators' schedule. Management, technical services, and contractor supervisory roles will work 5x2 schedule where appropriate.

As capital infrastructure is completed and handed over to the mine, the Owner's team will take over operations. This includes the shafts, UG dewatering and comminution circuits, and the drawpoint operations. Lateral and vertical capital development will continue under contractor support until the known LOM scope is completed. An Owner's team will then take over the care and maintenance, including rehabilitation, of UG workings.

Total required mining labour is summarized in Figure 16-70. This includes all on-site crews.

Figure 16-70: UG Labour Force



Source: JDS (2023)

16.11 Mine Equipment

The mobile equipment fleet for KDM is diesel-powered, trackless, and rubber tired. Mine development contractors will be utilized during pre-production and will be responsible for supplying all mobile equipment required for construction. KDM will take over mine development and operations at the conclusion of development contractor's scope and will purchase the required mobile mining fleet.

UG equipment requirements are built up based on the productivities (operating-hours) required for mining activities occurring within a given time period. As such, equipment requirements fluctuate throughout the mine life. Major equipment productivities used to estimate equipment requirements are as follows:

- Jumbo drilling: 75 m/hr;
- Longhole drilling: 12 m/hr;
- Bolter: 6-7 bolts/hour; and
- Mucking: 240 t/hr.

Peak equipment requirements for both the mine development contractor and Owner's team is summarized in Table 16-22.

Table 16-22: Mobile Equipment Requirements

Equipment	Peak Demand
Surface FEL (15 t / 5.4 m^3)	1
Surface Truck (39 t)	4
Surface Loader Crane	1
Surface Tractor	1
Surface Telehandler 24T	2
Surface Telehandler 10T	1
Surface Warehouse Forklift	1
LHD (7 t / 2.8 m^3)	1
LHD (17 t / 7.0 m^3)	4
LHD (21 t / 8 m^3)	3
Jumbo - 2 Boom	4
Longhole Drill - ITH	5
Secondary Breakage Drill	1
Bolter	3
Cable Bolter	2
Shotcrete Sprayer	2
Small Explosives Truck	2
Large Explosives Truck	1
Transmixer	2
Scissor Lift	3
Fuel/Lube Truck	1
Mechanics Truck	2
Electrician Truck	1
Boom Truck	1
Grader	1
Mobile Rock Breaker	1
Stationary Rock Breaker - 41 kW	2
Telehandler UG	3
Supervisor Truck	2
Utility Vehicle	3

Source: JDS (2023)

16.11.1 Electric Mine Fleet

A diesel LHD fleet should be used to initiate production activities at KDM until BEV technology is proven, and support services are established and available in Botswana. Diesel LHDs are a flexible and proven solution with low upfront capital costs and favourable economics. Issues with GHG emissions, working DPM exposure, and future increases to carbon pricing in diesel LHDs can be offset by introducing Tier 4 engines (which would require importing ultra-low sulphur diesel currently not offered in Botswana) or hybrid solutions.

BEV loaders can replace the planned diesel fleet at any time in the future with minimal change to the mining plan, as the electrical reticulation system has been designed with this future load in mind.

A tethered LHD fleet properly sized for KDM is not available and the smaller tethered LHDs which are available are not recommended in order to maintain operational flexibility, meet productivity targets, and reduce congestion on the extraction level. If a 21 t class tethered LHD fleet is developed, this would be a good alternative to be considered for future equipment replacement.

16.12 Mine Schedule

The shaft sinking schedule is provided by the shaft sinking contractor. JDS completed the remainder of the development and production schedule.

The project schedule looks forward from Q3 2023 and consists of a five-year pre-production period and a 13-year operating period.

The criteria used for scheduling the UG mine at KDM are as follows:

- The mine will operate two 12-hour shifts per day, 360 days per year;
- An average annual mill feed production rate of 2.7 Mt/a was scheduled, including ore from development and stopes; and
- Production ramp up over 8 months.

Shaft sinking commences din Q3 2021 and is completed by Q3 2026. Lateral development begins once the shaft sinking is complete. Production ramp up begins Q2 2027 with production commencing in Q4 2027.

16.12.1 Scheduling Philosophy

Both shafts are being sunk in parallel. Large scale lateral development which occurs after the shafts are sunk requires both shafts to be complete and a ventilation circuit established between the two. As such the sinking schedule has been developed to share in-shaft workloads and complete at the same time.

The two shafts rely on one another to achieve critical development milestones. Some of these include:

- Station connections at major shaft stations. As the V/S is not equipped with permanent dewatering lines all groundwater collected by the V/S will report to station sumps which pump out of the P/S;

- Development of fine ore bins. The top of the fine ore bins will be accessed by the V/S alone, and the bottom of the fine ore bins will feed conveyors which report to the P/S. In order for the P/S to be equipped, the V/S must provide access to the top of these bins such that they may be excavated and furnished;

- During P/S equipping the V/S will provide man and material access to the skip loading station for construction purposes; and

- During P/S equipping the shaft will be stripped of ventilation columns. All UG ventilation will rely on the ventilation columns within the V/S.

As the V/S is smaller than the P/S, and contains fewer services, it sinks at a faster rate than the P/S. The V/S will therefore be used to perform all lateral development between shafts at each station. The P/S will reach mine bottom prior to the V/S and will start equipping itself while the V/S makes final connections. At the completion of P/S equipping the V/S will be stripped of all its services, stage removed, headgear deconstructed, and replaced with the main fans for permanent exhaust operations.

Lateral development will commence on the 285 station where the shaft sinking contractor's lateral development equipment will be left. Initial priorities for UG development will include:

- Excavation, installation, and commissioning of the permanent dewatering pump station;

- Development to, and connection of, the 285 L flood drift and the 310 L Northern Drive towards the ore body, which will be connected by vertical raises to establish a ventilation circuit; and

- Ramp to P/S bottom, as to establish access for mucking out of shaft spillage.

With basic services of ventilation, water management, and waste management complete, the next priorities will include:

- Excavation, installation, and commissioning of the comminution circuit;

- Excavation and construction of extraction drives, crosscuts, and drawpoints;

- Development of 340 sub level and 380 drill horizon; and

- Excavation, installation, and commissioning of the workshops, refuge chambers, power stations, magazines, and other UG infrastructure.

Development priorities generally start at the bottom of the mine (285 L) and end at the top of the mine (670 L). As development fleets become available, they will be relocated to upper mine levels to develop production future production horizons.

16.12.2 Mine Development Schedule

Deswik scheduling software was used to optimize the mine development schedule. The shaft sinking schedule provided by sinking contractor was developed in Microsoft Project and then transferred into Deswik to combine the shaft sinking schedule with the development schedule.

16.12.2.1 Shaft Sinking Schedule

The following scheduling constraints apply to the shaft sinking program:

- Two shaft sinking crews. One per shaft:
  - 2.34 m/d V/S Sink; and
  - 1.98 m/d P/S Sink.
- One lateral development crew:
  - 3 days per machine to lower, assemble, and put to work; and
  - 65 m^3/d on all headings. Maximum two heading advance rates as available.
- One Shaft equipping crew:
  - 8.8 m/d equipping rate.

16.12.2.2 Lateral and Vertical Development Rates

The following scheduling constraints were used in Deswik for all lateral and vertical development:

- Maximum three development crews:
  - 3.5 m/d on priority headings, plus 2.75 m/d on auxiliary headings, to a maximum of 11 m/d per active jumbo.
- One raise boring crew:
  - 12 m/d Pilot;
  - 6 m/d Reaming; and
  - 92 m^3/d Slipping.
- Development capacity:
  - 1,000 m/month.

Lateral development is not able to commence until the shaft sink is fully complete.

The stope and development cycle times and productivities used for mine development and production scheduling were estimated from the first principles.

16.12.2.3 UG Infrastructure Installations

UG infrastructure installations have been accounted for within the mine schedule. Table 16-23 outlines the installation time budgeted for each major piece of UG infrastructure. A combination of contractors, equipment vendors, and Owner's team workforce will be utilized for infrastructure installations depending on the task and time period.

Table 16-23: Major Infrastructure Installation Durations

Infrastructure	Units	Duration (days)
1510 – Sub Station	Days	21
1510 – Rock Breaker	Days	129
1510 – Crusher Chamber	Days	184
1522 – Conveyor System	Days	206
1540 – Refuge (Portable)	Days	7
1550 – Typical Sump	Days	14
1558 – Pump Room	Days	74
1560 – Ventilation Bulkhead	Days	14
1560 – Ventilation Door	Days	30
1560 – Surface Fans	Days	154

Source: JDS (2019)

16.12.2.4 Mine Development Summary

Due to the mining method proposed at KDM the majority of the development needs to be complete before production can commence. In order to commence production stoping the shafts must be fully sunk and equipped, and all UG dewatering, ventilation, power distribution, and material handling systems need to be installed and commissioned. Sustaining mine development will consist of upper production levels not yet required to sustain the mine production rate, however, these levels will be developed as soon as possible to capitalize on the momentum the development contractor will have developed on site.

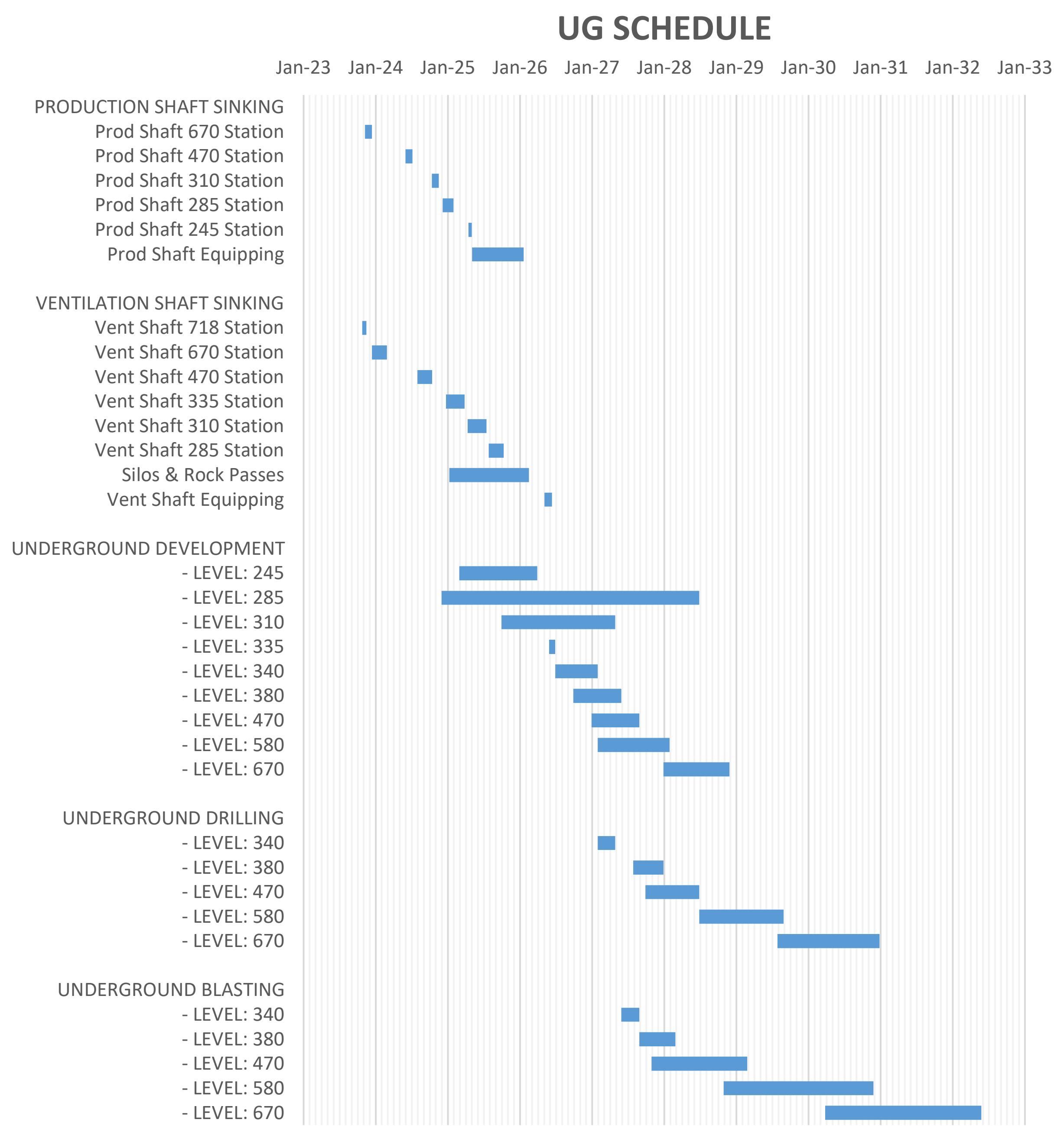
Mine development quantities are summarized in Table 16-24. Mine Development Schedule from Q3 2023 through to end of drill and blast is summarized in Figure 16-71.

Table 16-24: Mine Development Summary

Development	Units	Pre-Production	Sustaining	Total
Shaft Development	km	1.2	-	1.2
Lateral Development	km	16.3	3.5	19.8
Internal Raises	km	0.9	0.5	1.5
Total	**km**	**18.4**	**4.0**	**22.4**
	kt	**1,434**	**268**	**1,702**

Source: JDS (2023)

Figure 16-71: Mine Development Milestone Summary



Source: JDS (2023)

16.12.3 Mine Production Schedule

16.12.3.1 Production Rates

The following scheduling constraints were used in Deswik for all production activities:

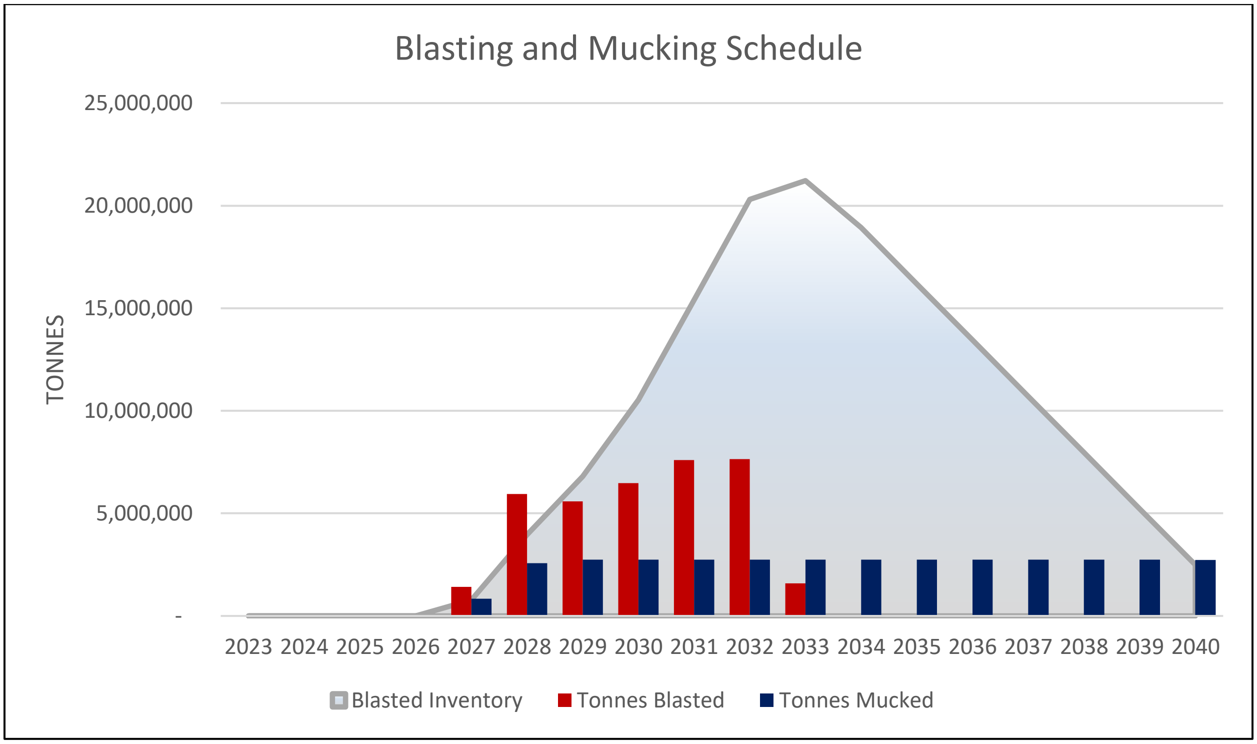
- Maximum of 105 m per shift per ITH drill;
- Maximum blasting rate of 21,000 t/d; and
- Maximum mucking rate of 216,000 t per month.

16.12.3.2 Mine Production Summary

Mine production of 7,400 t/d will be provided by draw down of the muck pile along with ore development during the production period.

Mine production commences in Q4 2027 six months after the first drawbell is blasted. Five ITH drills will be utilized to drill and blast approximately 21,000 t/d in order to supply 7,400 t/d of swell to the draw bells for the first six years of operations. Peak broken inventory occurs in year six (2033) for a total of 21 Mt. After six years, the South Lobe will be fully blasted, and mucking will continue at a constant rate of 7,400 t/d until the UG reserves are depleted at the end of year 13 (2040). Figure 16-72 illustrates the relationship between the blasted inventory and mucked inventory over time.

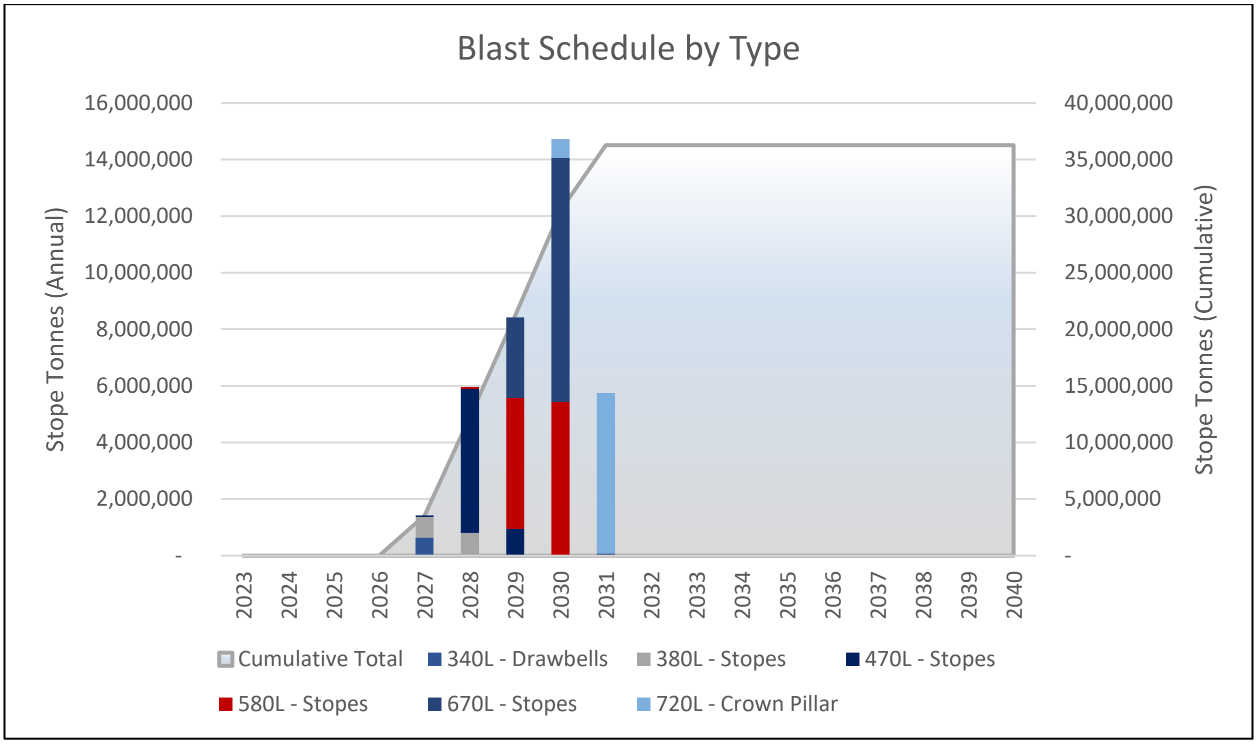
Figure 16-72: Blasting and Mucking Schedule Summary



Source: JDS (2023)

Figure 16-73 illustrates the blasting of the different production stope types over time. Of notable interest is the wrecking of the crown pillar, planned to commence in 2030.

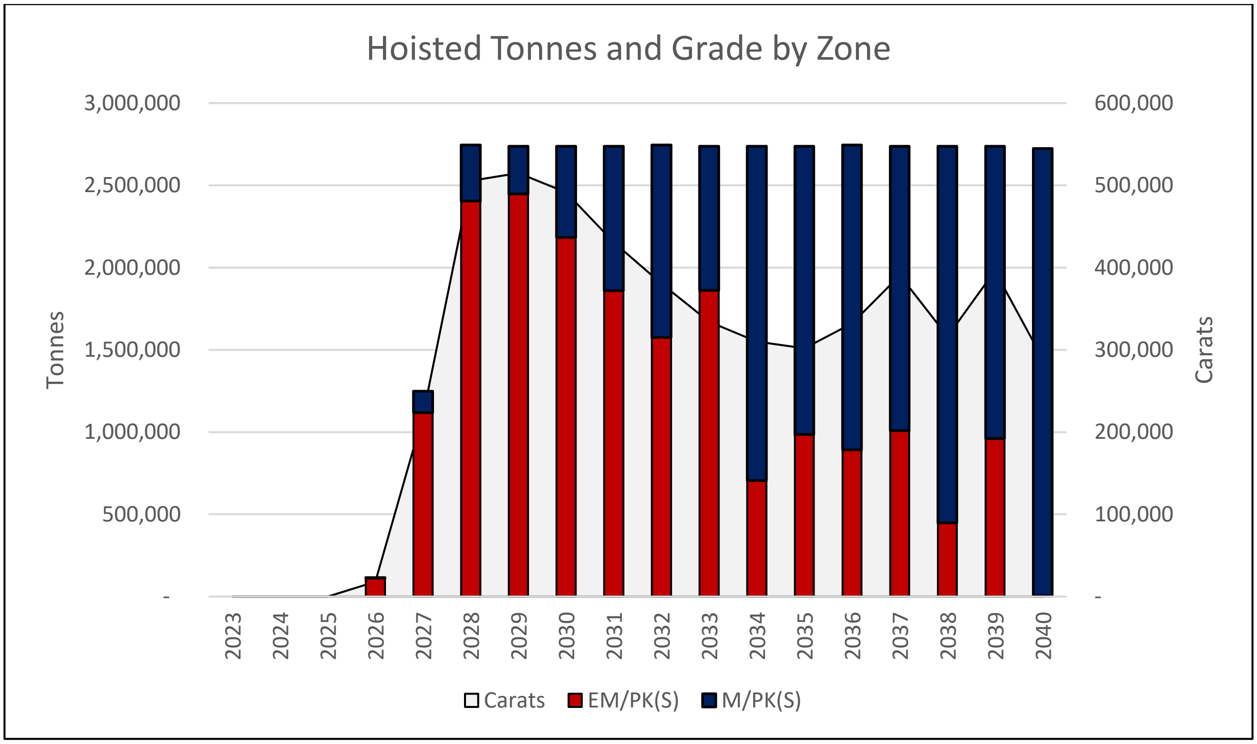
Figure 16-73: Blasting Schedule by Stope Type



Source: JDS (2023)

Figure 16-74 illustrates the breakdown between mineralized zones over time.

Figure 16-74: Hoisted Tonnes and Grade by Domain



Source: JDS (2023)

16.12.4 UG Production Schedule

A number of schedule iterations and manual adjustments were made to produce a robust, sensible, and realistic schedule.

Final results of the Deswik schedule were organized such that physical metres, tonnes and carats were broken down into different categories for direct use in the cost model. From the final schedule, cost model requirements including items such as the mining fleet, workforce, consumables, ventilation, pumping, and power were determined and used to develop costs from first principals. Reports were generated monthly and then summed into annual totals for financial modelling.

The annual mine production schedule provided in Figure 16-75 shows annual summaries of ore and waste tonnage mined, ore grades, and carats. Ore and waste tonnages have been rounded to the nearest million.

Schedule is forward looking from Q3 2023.

Table 16-25: Summary of UG Mining

Parameter	Unit	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Summary of Development																				
Shaft Development	km	1.2	0.3	0.7	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lateral Development	km	19.8	0.1	0.5	1.4	7.5	7.3	3.0	-	-	-	-	-	-	-	-	-	-	-	-
Internal Raises	km	1.5	-	-	0.2	0.2	0.6	0.5	-	-	-	-	-	-	-	-	-	-	-	-
Metres per month per jumbo	m/m/jumbo	139	55	55	117	204	185	125	-	-	-	-	-	-	-	-	-	-	-	-
Lateral Daily Advance	m/d	21	-	1	4	21	20	8	-	-	-	-	-	-	-	-	-	-	-	-
Summary of Drill and Blast																				
Development Ore	Mt	0.7	-	-	-	0.1	0.4	0.2	-	-	-	-	-	-	-	-	-	-	-	-
Drawbells	Mt	0.6	-	-	-	-	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-
Stoping	Mt	29.3	-	-	-	-	0.8	6.0	5.6	6.5	7.6	2.9	-	-	-	-	-	-	-	-
Crown Pillar	Mt	6.3	-	-	-	-	-	-	-	-	-	4.7	1.6	-	-	-	-	-	-	-
Total Blasted	Mt	37.0	-	-	-	0.1	1.8	6.1	5.6	6.5	7.6	7.6	1.6	-	-	-	-	-	-	-
Drill and Blast Rate	kt/d	19.5	-	-	-	0.3	5.0	16.8	15.3	17.7	20.8	20.9	4.4	-	-	-	-	-	-	-
Summary of Inventories																				
Drilled Inventory	Mt	12.6	-	-	-	-	2.1	4.9	8.3	11.1	12.6	8.6	1.0	-	-	-	-	-	-	-
Blasted Inventory	Mt	21.2	-	-	-	-	0.8	4.0	6.8	10.5	15.4	20.3	21.2	18.9	16.2	13.4	10.7	8.0	5.2	2.5
Summary of Production																				
Hoisted Ore	Mt	37.0	-	-	-	0.1	1.2	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7
EM/PK(S)	Mt	18.6	-	-	-	0.1	1.1	2.4	2.4	2.2	1.9	1.6	1.9	0.7	1.0	0.9	1.0	0.4	1.0	-
M/PK(S)	Mt	18.4	-	-	-	-	0.1	0.3	0.3	0.6	0.9	1.2	0.9	2.0	1.8	1.9	1.7	2.3	1.8	2.7
Hoisted Grade	cpht	14.2	-	-	-	15.8	17.7	18.4	18.8	18.0	15.7	13.9	12.2	11.3	11.0	12.1	14.3	11.6	14.5	10.6
EM/PK(S)	cpht	18.1	-	-	-	16.0	18.6	19.7	19.9	20.0	18.6	17.1	13.7	15.1	12.2	14.4	19.9	21.0	22.6	22.9
M/PK(S)	cpht	10.2	-	-	-	9.3	9.8	9.6	10.0	9.9	9.6	9.4	9.0	10.0	10.4	11.0	11.0	9.7	10.0	10.6
Hoisted Carats	kc	5,232	-	-	-	18	221	505	515	491	431	380	334	310	302	332	391	316	396	289
EM/PK(S)	kc	3,361	-	-	-	18	208	473	486	436	346	270	256	106	120	129	201	94	217	1
M/PK(S)	kc	1,871	-	-	-	-	13	33	29	55	84	111	79	204	182	204	190	222	179	288
Hoisted Waste	Mt	984	38	111	128	484	162	61	-	-	-	-	-	-	-	-	-	-	-	-

Source: JDS (2023)

16.12.5 Combined OP and UG Production Schedule

The OP and UG mine production schedule for KDM incorporates Centre lobe reserves mined from the OP, and the South lobe reserves mined from both OP and UG operations. The mill feed will be provided from the OP and the existing stockpiles, until the UG reaches commercial production at the end of 2027. The OP will operate until mid-2025; the existing surface stockpiles will be consumed as processing capacity comes available. OP and UG material will be stockpiled as needed when mine production exceeds mill capacity.

The Lucara Botswana mining technical services team has provided the OP production targets and mine plan. After the completion of the OP in mid-2025, and with the introduction of early development ore in mid-2026, the combined OP and UG production schedule deviates from the existing OP only schedules.

The mill blending and stockpiling strategy after the completion of the OP in mid-2025 is based on the following criteria:

- Mill feed is prioritized based on value / tonne; and
- UG feed is a mix of EM/PK(S) and M/PK(S) as UG material handling operations do not allow for selectivity between ore domains.

Table 16-26 summarizes the combined LOM production schedule for KDM, including the OP and UG mines, the mill feed schedule, and stockpile balances. Note that 2023 values are a combination of actuals from Q1 and Q2 2023 and forecasts from the latter half of the year.

Table 16-26: Combined LOM Production Schedule

Description	Unit	Total	Year Summary																			
			2023[1]	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Mining Summary																						
Waste - OP Mining	Mt	2.6	1.5	1.1	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore - OP Mining	Mt	6.7	2.6	3.0	1.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore - UG Mining	Mt	37.0	-	-	-	0.1	1.2	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	-	-
Mill Feed																						
Direct Feed	Mt	41.8	1.9	2.6	1.0	0.1	1.2	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	-	-
From Stockpiles	Mt	11.8	1.0	0.1	1.8	2.6	1.5	-	-	-	-	-	-	-	-	-	-	-	-	-	2.7	2.0
Total Mill Feed	Mt	**53.6**	**2.8**	**2.8**	**2.8**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.0**
	cpht	13.1	14.2	13.2	15.4	12.2	11.6	18.6	18.9	18.1	15.8	13.9	12.3	11.3	10.9	12.0	14.5	11.6	14.5	10.5	5.7	4.5
	000's ct	7,026	403	365	428	331	313	503	510	489	427	376	333	304	295	325	390	314	392	284	154	91
Mill Feed - By Domain																						
North	Mt	0.5	0.1	-	-	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	13.5	12.2	-	-	13.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	000's ct	62	11	-	-	51	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Centre	Mt	3.5	0.6	0.2	1.0	1.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	14.7	17.6	15.0	16.6	12.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	000's ct	517	105	33	159	220	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP-South-EM/PK(S)	Mt	1.5	0.5	0.6	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	25.0	23.1	24.5	27.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	000's ct	372	114	140	118	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP-South-M/PK(S)	Mt	5.4	1.7	2.0	1.4	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	10.1	10.5	9.7	10.8	8.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	000's ct	547	173	192	151	31	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mixed	Mt	5.8	-	-	-	0.1	1.5	-	-	-	-	-	-	-	-	-	-	-	-	-	2.2	2.0
	cpht	5.1	-	-	-	8.9	6.6	4.5	-	-	-	-	-	-	-	-	-	-	-	-	4.5	4.5
	000's ct	296	-	-	-	9	97	-	-	-	-	-	-	-	-	-	-	-	-	-	99	91
UG-South-EM/PK(S).	Mt	18.6	-	-	-	0.1	1.1	2.5	2.4	2.2	1.9	1.6	1.9	0.7	1.0	0.9	1.0	0.4	1.0	-	-	-
	cpht	18.1	-	-	-	15.9	18.6	19.6	19.9	20.0	18.6	17.1	13.7	15.2	12.1	14.5	20.1	20.6	22.6	22.9	-	-
	000's ct	3,361	-	-	-	19	203	481	481	436	345	270	255	109	119	130	199	91	221	1	-	-
UG-South-M/PK(S)	Mt	18.4	-	-	-	-	0.1	0.2	0.3	0.5	0.8	1.1	0.8	2.0	1.7	1.8	1.7	2.3	1.7	2.7	0.5	-
	cpht	10.2	-	-	-	9.6	10.0	8.7	10.3	10.2	9.7	9.4	9.2	9.8	10.2	10.8	11.2	9.9	9.9	10.5	10.7	-
	000's ct	1,871	-	-	-	-	13	22	29	53	82	106	78	195	176	195	191	223	171	282	55	-

Description	Unit	Total	Year Summary																			
			2023[1]	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Stockpile Inventory – Start of Period																						
North																						
HG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	0.4	0.4	0.4	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Centre		-																				
HG	Mt	-	1.1	0.9	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	1.7	1.7	1.7	1.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP - South-EM/PK(S)																						
HG	Mt	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP - South-M/PK(S)																						
HG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LG	Mt	-	0.9	0.9	0.9	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UG																						
EM/PK(S) + M/PK(S)	Mt	-	-	-	-	-	-	-	-	0.1	0.1	0.2	0.2	0.2	0.3	0.3	0.4	0.4	0.4	0.5	0.5	-
Life of Mine																						
Contact	Mt	-	0.7	0.8	0.8	0.8	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LOM	Mt	-	4.7	5.0	5.0	5.0	5.0	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	2.0
Total Stockpile	Mt	-	**9.6**	**9.7**	**10.0**	**8.3**	**5.7**	**4.2**	**4.2**	**4.3**	**4.3**	**4.4**	**4.4**	**4.4**	**4.5**	**4.5**	**4.6**	**4.6**	**4.6**	**4.7**	**4.7**	**2.0**

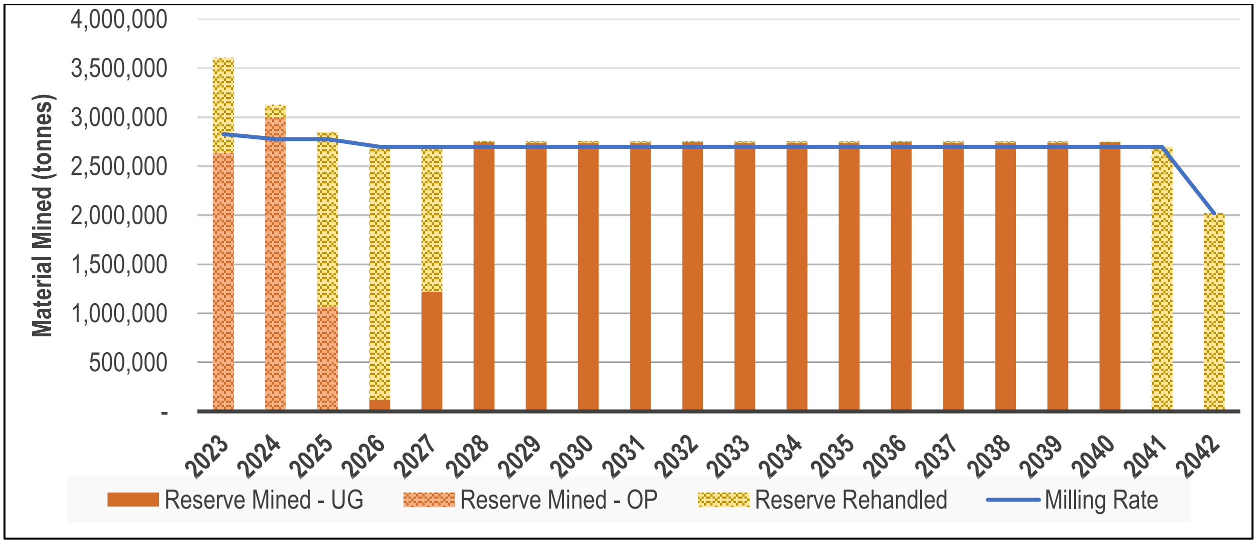
Notes:

1. 2023 values are a combination of actuals from Q1 and Q2 2023 and forecasts from the latter half of the year. Mineral Reserve Estimation is of June 30, 2023.

Source: JDS (2023)

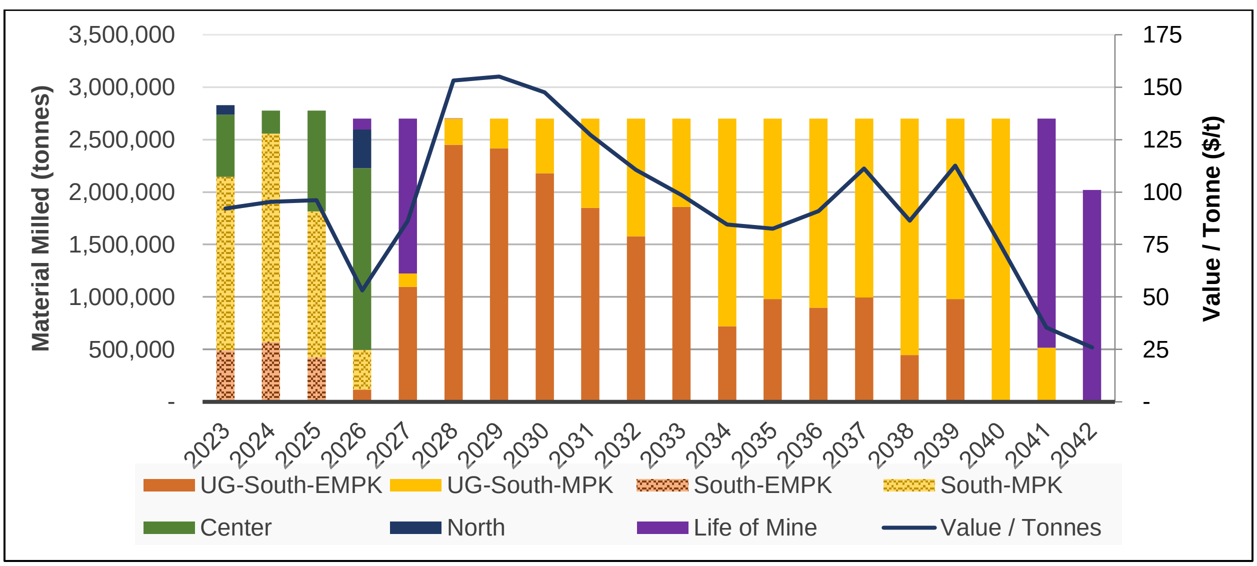
The total blended mine and mill feed from both UG, OP, and stockpile operations is show in Figure 16-75 and Figure 16-76.

Figure 16-75: Summary of Mine Production



Source: JDS (2023)

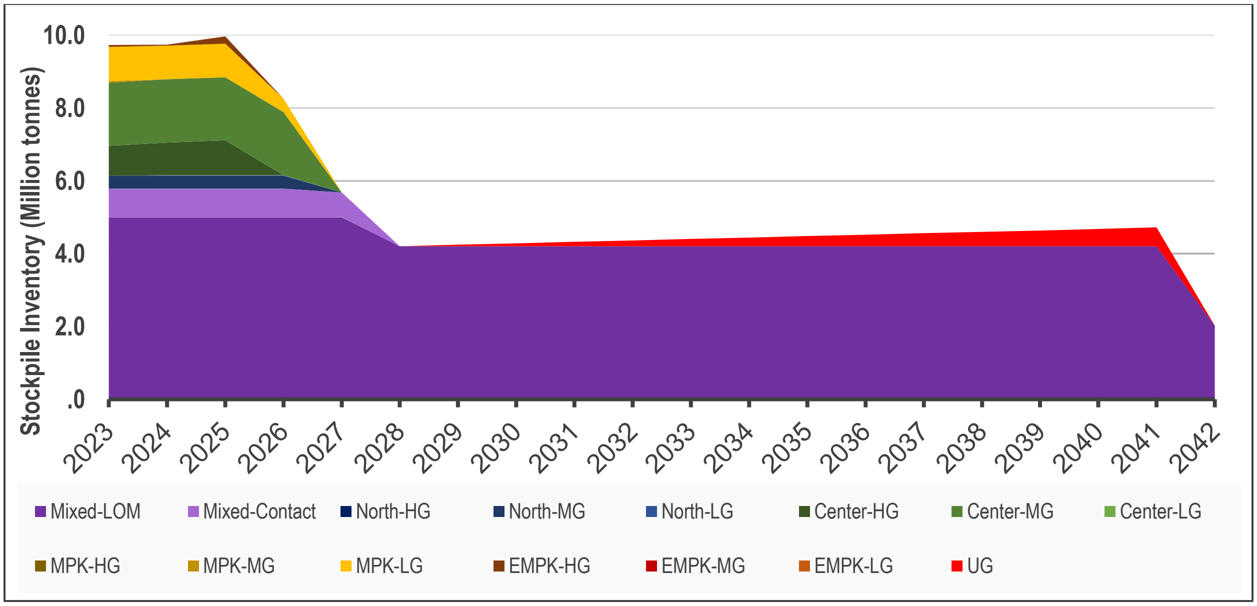
Figure 16-76: Summary of Mill Production



Source: JDS (2023)

A summary of the stockpile inventory opening balance is outlined in Figure 16-77.

Figure 16-77: Summary of Stockpile Inventory Opening Balance



Source: JDS (2023)

17 PROCESS DESCRIPTION / RECOVERY METHODS

17.1 Introduction

DRA Projects Pty Ltd. (DRA) was commissioned by JDS Energy & Mining Inc. (JDS) on behalf of Lucara Diamond Corp. (Lucara) to perform an overall treatment plant evaluation as part of a Feasibility Study (FS) on extending the life of KDM by mining underground (UG) after the completion of open pit (OP i.e., surface) mining.

To successfully assess current plant performance and production, the following Lucara Botswana employees were engaged and consulted to source the desired information and data as part of the overall treatment plant evaluation:

- Lucas Ntsipe, Lucara Botswana Assistant General Manager;
- Bailey Maila, Lucara Botswana (Acting) Process Manager;
- Glen Wright, Lucara Botswana Plant Metallurgist;
- Tiroyaone Kesiilwe, Lucara Botswana Senior Process Engineer – Technical; and
- Catherine Mrosso, Lucara Botswana Acting Production Superintendent – Wet End.

The following sub sections provide a brief historical summary associated with KDM since its inception in 2012.

17.1.1 KDM Phase I (Greenfields) History

Boteti Diamonds (a subsidiary of Lucara Diamond Corporation at that stage) contracted DRA Mineral Projects to provide complete EPCM services for the design and construction of a milling, Dense Media Separation (DMS), Recovery Plant, associated crushing, screening and thickener systems for KDM (called AK06 Mine at that time).

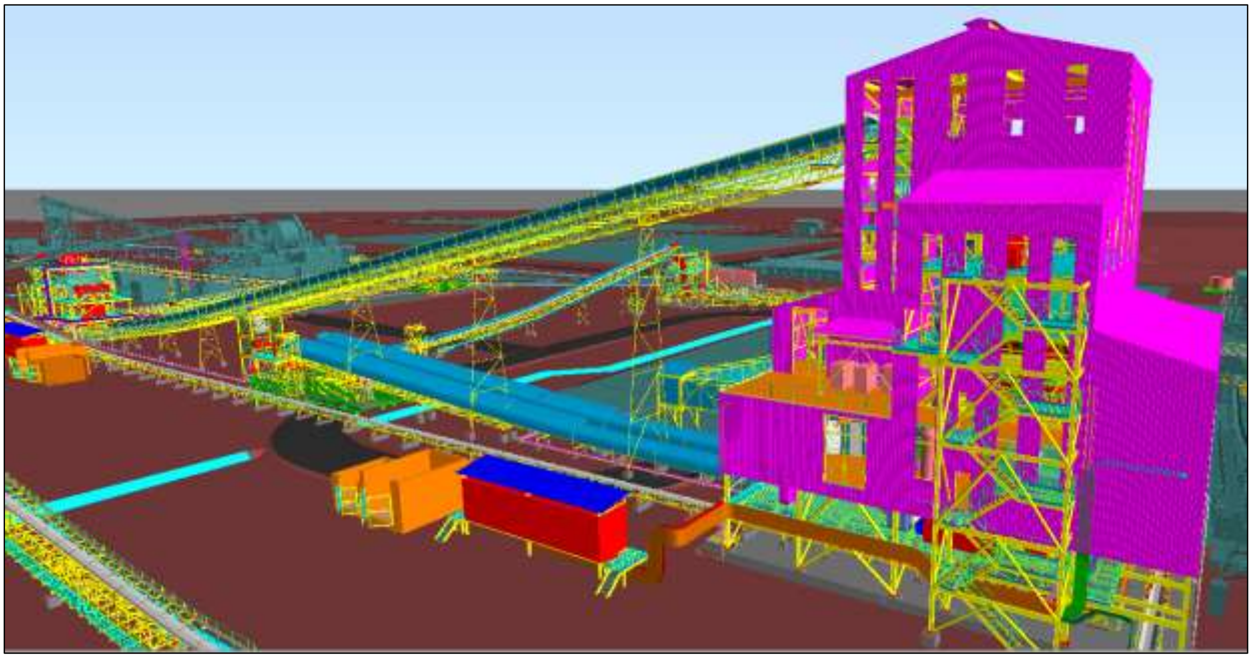
KDM was designed to process 2.5M t of Run of Mine (ROM) kimberlite ore per annum with a single 200 t/h DMS module. The concentrate material from the DMS was subsequently treated through a 2.5 t/h wet X-ray Recovery for material reduction and diamond winning. Adequate space was allowed for during the Phase I layout design to make provision for future plant expansions – in particular around the milling and DMS sections.

A unique feature of KDM during Phase I was the autogenous (or AG) milling technology utilized as part of the circuit: previously seen predominantly only in northern hemisphere diamond plants. AG mills can accomplish the same size reduction work that normally takes multiple stages of crushing, screening and grinding methods which accounts for its popularity. It also lends itself to high volume processing. The treatment plant and Recovery were successfully commissioned in April 2012.

17.1.2 KDM Phase II (Brownfields) History

The brownfields Phase II KDM Plant Upgrade Project was an expansion of the Phase I Greenfields AG Mill plant to cater for large diamond recovery upfront in the circuit ahead of the DMS.

Figure 17-1: Model View of KDM's Phase II XRT Section



Source: DRA (2015)

With regards to the Phase II expansion completed in 2015, EPCM services were provided for the design, construction and commissioning of a new secondary (gyratory) crushing, XRT sizing, and XRT diamond recovery circuits.

A unique feature/aspect of the KDM Phase II project was the utilization of XRT machines in a large diamond recovery circuit to recognize and recover carbon-signature material (i.e., diamonds). By employing this technology in the process treatment plant, the top cut-off size of the plant could be significantly increased allowing for large stones to be recovered where previously they would have been broken in the pebble crusher and mill. In addition, XRT mitigated the impact of the high density of the KDM kimberlite on the DMS performance as the DMS was limited to treating -8 mm material only.

What made the KDM Phase II project even more unique is the fact that XRT was also utilized in an audit function: where a portion of the -20 +8 mm tails from the main XRT building was treated through a single 50 t/h capacity downstream sorter for both metallurgical accounting and scavenging purposes.

Figure 17-2: Construction Completed and Fully Commissioned KDM Phase II XRT Building

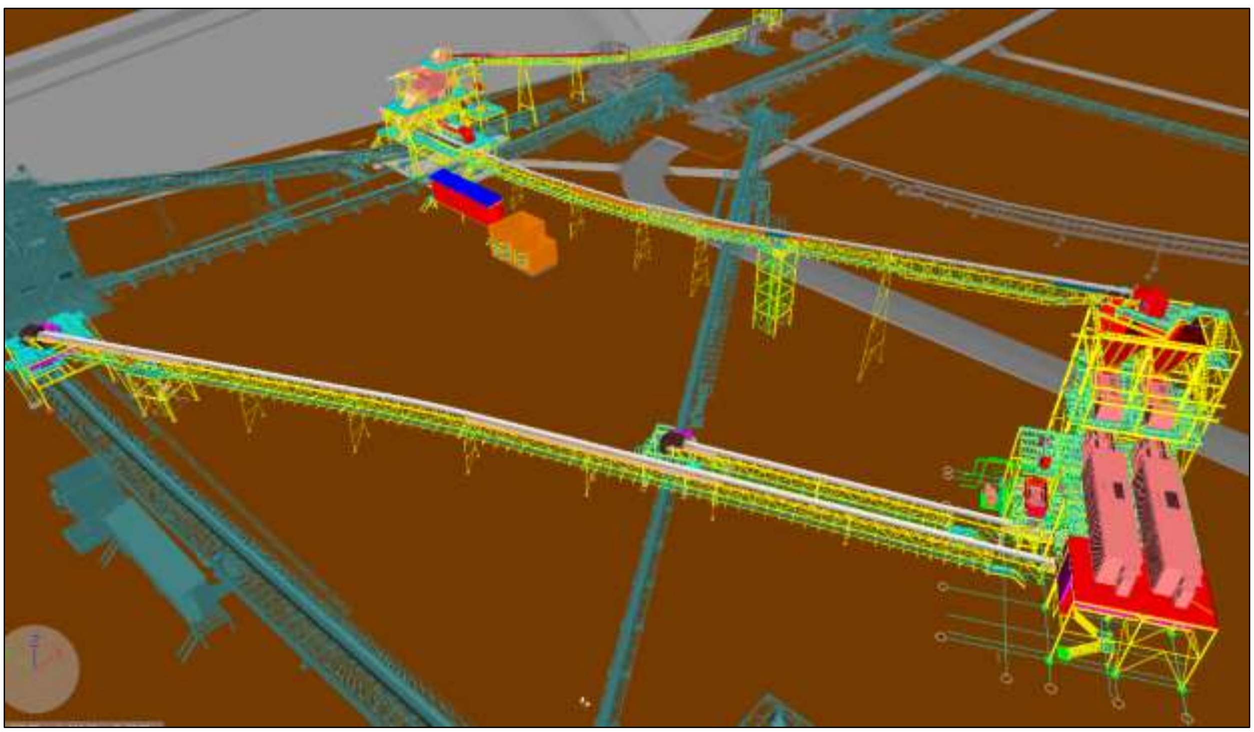
Source: DRA (2015)

17.1.3 KDM Mega Diamond Recovery and Phase III (Brownfields) History

The Brownfields Mega Diamond Recovery (MDR) Project was a Lump Sum Turnkey (LSTK) addition to the Phase II KDM Expansion Project, allowing for the inclusion of XRT sorting technology ahead of the AG Mill with the aim of sterilizing the feed of liberated mega diamonds above 50 mm by adding a recovery step upfront which was only top size limited by the available technology. A unique feature/aspect about the MDR Project was that it was the largest top size cut of any diamond plant known in the industry at the time, with sorting conducted on material passing 125 mm prior to AG Mill comminution.

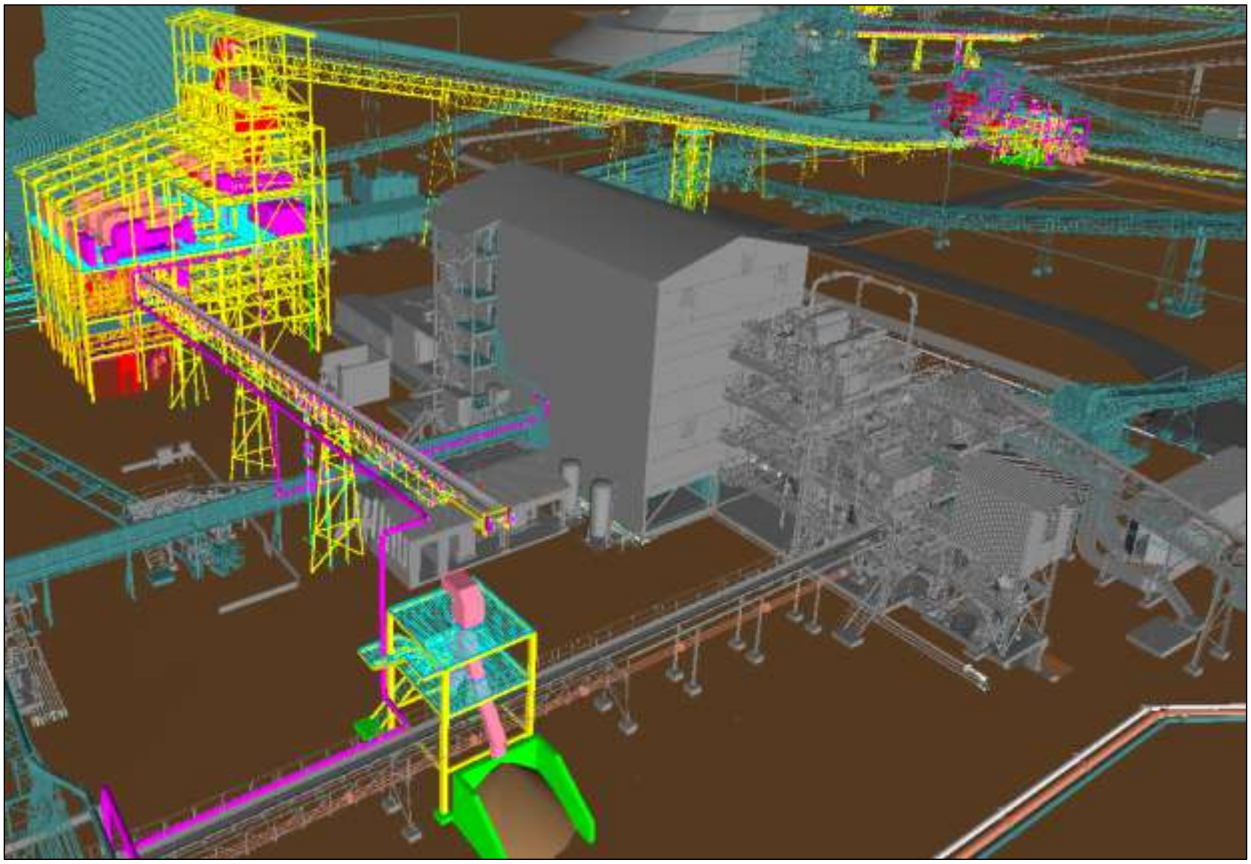
The Brownfields Phase III KDM Plant Upgrade Project was another supplementary expansion to the KDM Phase II Expansion Project, providing complete EPCM services for the design, construction, and commissioning of the Phase III brownfields expansion. Phase III made provision for the inclusion and application of XRT sorting technology to the 4 x 8 mm size fraction ahead of the DMS – with the ultimate aim of negating the high-frequency near density content of KDM's Unit 13 (M/PK(S)) ore which could result in DMS yields in excess of ~25 %. A unique feature/aspect associated with this Project was that it was the smallest fraction of XRT bulk sorting technology applied on a diamond mine (at that time) between the 4 and 8 mm size range. This is required due to the unique variance in ore body characteristics at KDM; which has yielded some of the biggest diamonds in history – whilst at the same time having to negotiate one of the highest density and hardest kimberlites in existence.

Figure 17-3: KDM MDR Project – 3D Model Snapshot



Source: DRA (2017)

Figure 17-4: KDM Phase III Model Showing Primary XRT Machines



Source: DRA (2017)

17.2 Plant Design Criteria

The following KDM Process Design Criteria (PDC) presented below is a high-level summary from predominantly the Phase I and II design and built.

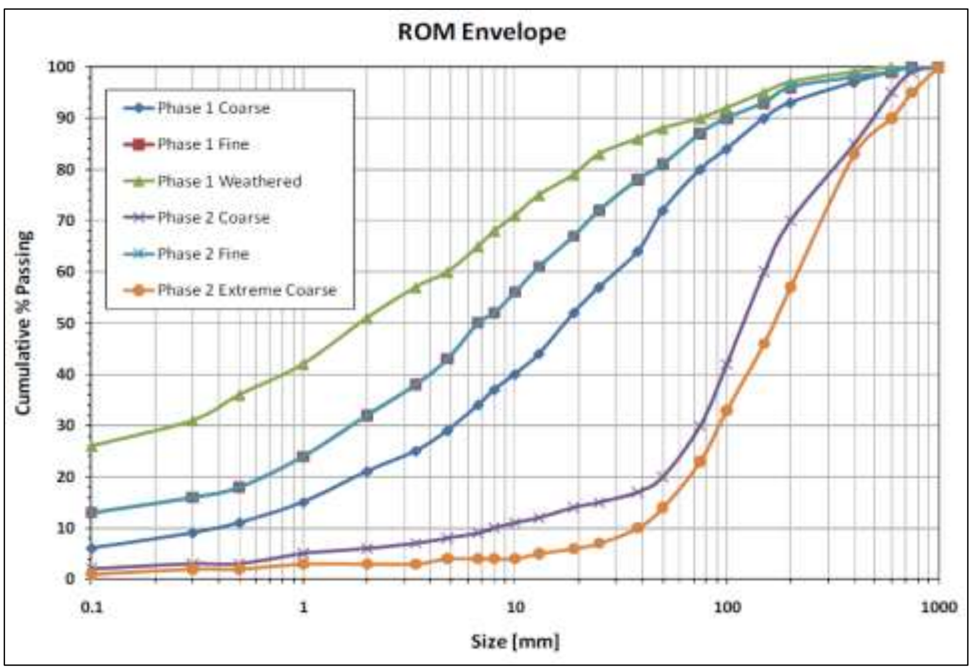
The following source codes are used to reference the origin of each item of information that appears in the design criteria.

Table 17-1: Process Design Criteria Source Codes

Code	Description
D1	Selected by DRA, based on design requirements
D2	Selected by DRA, based on testwork data
D3	Selected by DRA, based on other inputs
A	Assumed
C	Specified by Client
V	Information by vendors or third parties

Source: DRA (2014)

Table 17-2: Process Design Criteria

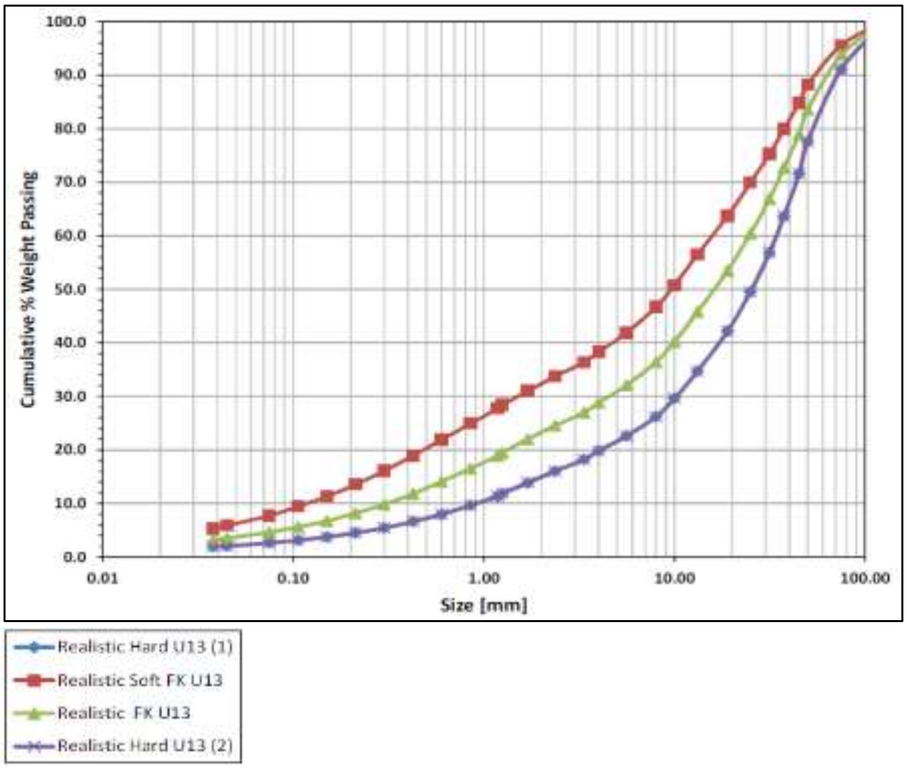
Criteria	Units	Value	Source	Revision
Ore type to be treated	-	Diamond bearing kimberlite	C	A
Design annual tonnage	dry mt/a	2.5 - 3.5	C	0
Manned hours per annum	hrs pa	8,760	C	A
Overall utilisation	%	81.0	D3	A
"On ore" hours per year	hrs pa	7,095	D3	A
Design throughput	t/h	350 - 500	C	A
Operation type	-	Continuous	C	A
Top cut off size	mm	60.0	D3	A
Bottom cut off size	mm	1.5	C	A
ROM moisture content	wt %	8.0	A	A
Clay mineral content	%	3.0	A	A
Run Of Mine Particle Size Distributions				
			V	0

Note: "Phase 2" in graph above denotes unweathered M/PK(S) kimberlite ore type

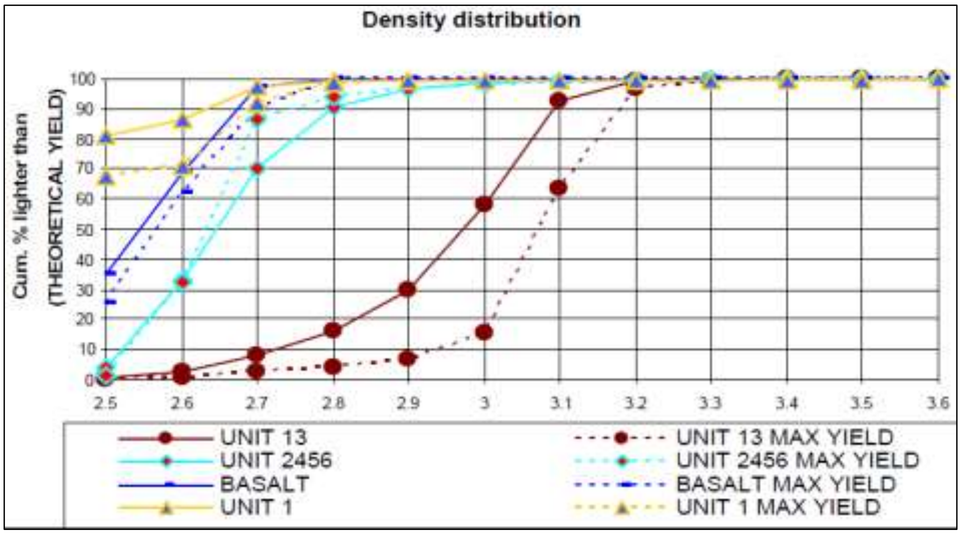
Criteria	Units	Value	Source	Revision
Crushability Data				

DESCRIPTION		T_{10} [%] @0.25 kWh/t	T_{10} [%] @1 kWh/t	Source	Revision
Unit 1	Average	32	56	V	0
	Min	18	34		
	Max	63	71		
Unit 2456	Average	20	51		
	Min	14	40		
	Max	31	62		
Unit 8	Average	12	38		
	Min	10	36		
	Max	14	41		
Unit 13	Average	14	37		
	Min	11	31		
	Max	16	46		
Unit 13 (JK Tech DWT, August 2010)	Average	8	24		
	Min	4	19		
	Max	12	31		
Unit 14	Average	10	35		
	Min	8	31		
	Max	12	38		
Basalt	Average	15	50		
	Min	14	47		
	Max	16	52		
MDSTN	Average	12	27		
	Min	9	24		
	Max	17	31		

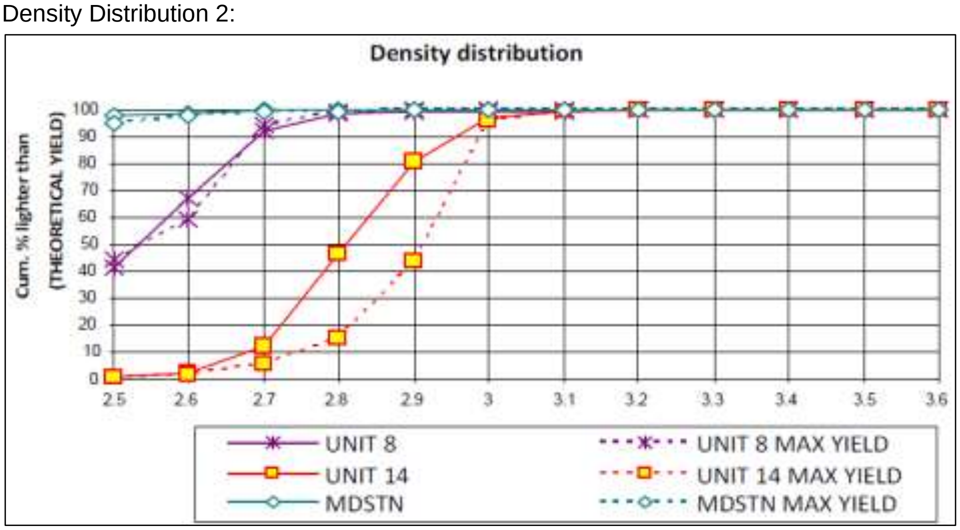
Criteria	Units	Value	Source	Revision
Secondary Comminution				
Pre-crusher split	%	0 - 100	D3	A
Scalping screen cut size	mm	60.0	D3	A
Pre-crusher feed F100	mm	300	V	A
Crusher Type	-	Secondary Gyratory	D1	A
Closed side setting	mm	60 - 75mm	D3	A
AG Milling				
Discharge grate	-	TPL type grate	D2	A
Circuit Feed Size (Fresh Feed): F80	mm	~125.0	D2	A
Circuit Product Size: P80	mm	~37.5 - 50.0	D2	A
Circuit Product Size: % -1.5mm	%	~13 - 30	D1	A
Pinion Power (Mill Power)	kW	~3 045 - 3 783	D1	0
Installed Power	kW	4 000	D1	A
Mill Speed (Critical RPM)	RPM	14.6	D1	0
Mill Speed (% Nc)	% Nc	~80 - 82	D1	A
Circulating Load (% of Fresh Feed)	%	~5.5 - 12.5	D1	A
In Mill Density	% (v/v)	~68 - 70	D1	A
Product Slurry Density Target (-1.5mm, before dilution)	t/m^3	1.09	D1	A

Criteria	Units	Value	Source	Revision
Product Slurry Density Target (-1.5mm, before dilution)	% (w/w)	12.8	D1	A
Expected Mill Product Particle Size Distributions				
			D2	0
Pebble Crusher and Bleed Screen				
Pebble Crusher Closed side setting	mm	25.0	D3	A
Bleed Screen Cut size	mm	32.0	D3	A
-32mm mill bypass	%	0, 12.5, 25, 37.5, 50, 62.5, 75, 100	D3	A
XRT Bulk Sorters				
Technology	-	XRT	D2	A
Size fraction: Middles	mm	8 - 14	D1	A
Size fraction: Coarse	mm	14 - 32	D1	A
Size fraction: Large	mm	32 - 60	D1	A
Diamond recovery (Large, Coarse % Middles)	%	≥ 98	C	A
Fines DMS				
Feed size	mm	1.5 - 8	D1	A
De-rated throughput	t/h	150 - 200	D1	A

Criteria	Units	Value	Source	Revision
Expected yields				
Average	%	7.40	D1	A
75th percentile	%	11.1	D1	A
Density Distribution 1:			V	0

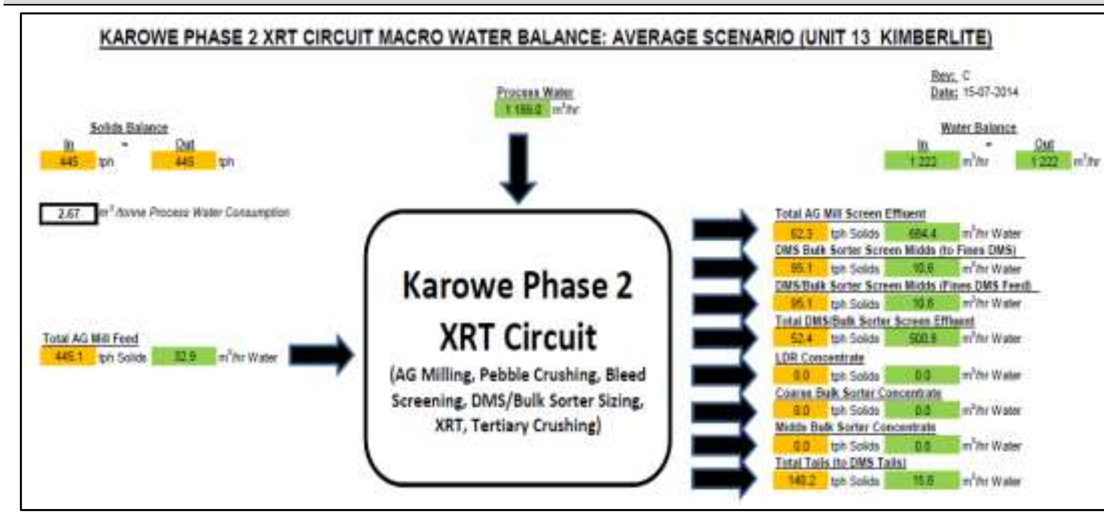


Criteria	Units	Value	Source	Revision
Density Distribution 2:			V	0



Criteria	Units	Value	Source	Revision
Recovery Plant (Phase 2)				
Feed size	mm	1.5 - 8	D1	A
Expected yield - Average	t/h	9.80	D1	A
Expected yield - 75% Percentile	t/h	15.4	D1	A
DMS Concentrate Size Distribution				

Criteria	Units	Value	Source	Revision
-8 +4mm	%	60.0	D2	A
-4 +1.5mm	%	40.0	D2	A
Average Yield Throughput				
+4mm	t/h	5.88	D1	A
+1.5mm	t/h	3.92	D1	A
75 Percentile Throughput				
+4mm - Middles	t/h	9.24	D1	A
+1.5mm - Fines	t/h	6.16	D1	A
Wet MagRoll Capacity (Based on 2 Streams)				
+4mm - Middles 5 t/h	t/h	10.0	V	A
+1.5mm - Fines 3 t/h	t/h	6.00	V	A
MagRoll Reduction	%	65.0	D2	A
Wet X-ray Capacity (Based on 2 Streams)				
+4mm - Middles 1950 kgh	t/h	4.00	V	A
+1.5mm - Fines 1050 kgh	t/h	2.00	V	A
Feed to X-ray Circuit				
Average Yield +4mm - Middles	t/h	2.06	D1	A
Average Yield +1.5mm - Fines	t/h	1.37	D1	A
75 Percentile Yield +4mm - Middles	t/h	3.23	D1	A
75 Percentile Yield +1.5mm - Fines	t/h	2.16	D1	A
Reconcentration X-ray Capacity				
+4mm - Middles	kgh	25.0	V	A
+1.5mm - Fines	kgh	10.0	V	A
Phase 2 Macro Water Balance				

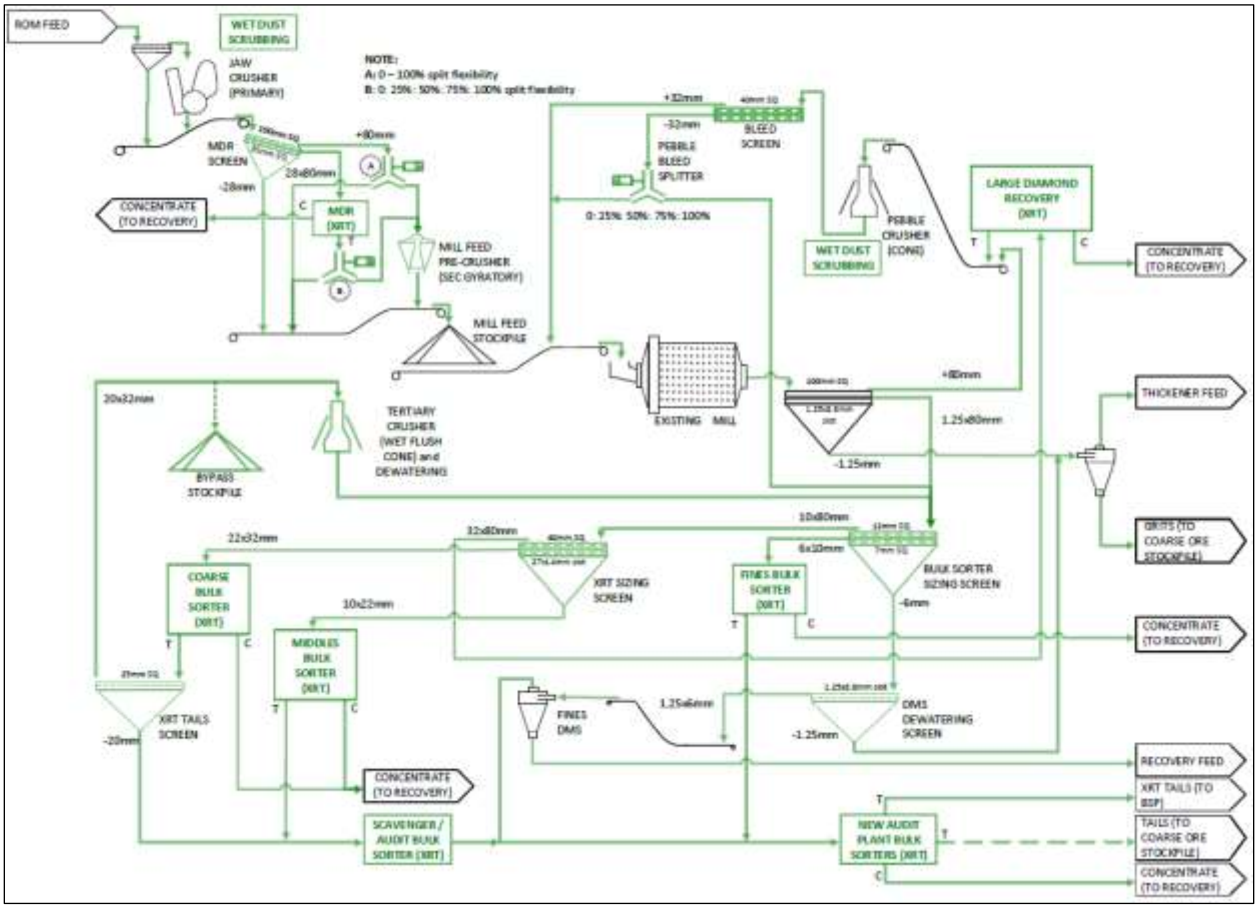


0

Source: DRA (2014)

17.3 Plant Design and Current Plant Performance

Figure 17-5: Overall KDM Block Flow Diagram (Current)



Source: DRA (2019)

Figure 17-5 has been updated to include all previous inception and subsequent expansion phases, as well as plant upgrades, presenting a general overview in block flow format of the current KDM treatment plant process highlighting mainstream flows, products and by-products. The equipment items highlighted in black font denotes the original kit from Phase I, whilst the equipment and streams highlighted in green font denote subsequent changes post the Greenfields Phase I built. A high-level process description for mainstream areas can be found further down in this section. ROM ore currently fed to the process treatment plant is that of Magmatic Pyroclastic Kimberlite (M/PK(S)) – which is considered a more competent, harder ore type; and Eastern Magmatic Pyroclastic Kimberlite (EM/PK(S)) – seen as a competent, higher grade ore type.

A list of major equipment duties currently in existence and functioning as part of the KDM treatment plant process flowsheet, can be viewed in Table 17-3 below. The tabulated summary list includes all key equipment duties with installed drives noticeably equal to or larger than 100 kW; spanning from first treatment plant construction and commissioning in 2012 and covers all three phases of Greenfields first-built and Brownfields expansion projects.

Table 17-3: List of Major Components – Summary Mechanical Equipment List for Installed Drives ≥ 100 kW

Tag Number	Description	Specification	Installed Power (kW)
100-CJA-045	Primary Jaw Crusher	Size: CJ613	160
120-FCV-005	In Plant Stockpile Feed Conveyor	Width: 1200 mm	220
200-AGM-010	AG Mill	Size: 8.53 m Ø diam x 4 m long	4 000
200-PCB-030	AG Mill - Effluent Pump	Size: 10/8F-AH-5VCM	160
220-CCA-020	Pebble Crusher	Size: XL 400 Excel-Raptor (cone crusher)	300
300-PCB-045	Cyclone Feed Pump	Size: 10/8F-AH-5VCM	250
300-PCB-120	CM Pump	Size: 10/8F-AH-5VCM	160
500-PCB-090	Slimes Disposal Pump No. 1	Size: 8/6F-AH-6VCM (High Efficiency)	132
500-PCB-095	Slimes Disposal Pump No. 2	Size: 8/6F-AH-6VCM (High Efficiency)	132
500-PCB-100	Slimes Disposal Pump No. 3	Size: 8/6F-AH-6VCM (High Efficiency)	160
520-PCC-025	Mill Process Water Supply Pump	Size: NF200-500-P55	185
115-GGA-035A	Secondary Crusher	Model: KG4513 (Secondary Gyratory)	185
380-CCA-030A	Tertiary Crusher	Model: Cybas-i 1200 (wet flush cone crusher)	220
520-PCC-200	DMS/Bulk Sorter Process Water Pump	Size: NF200-400-P55	110

Source: DRA (2015)

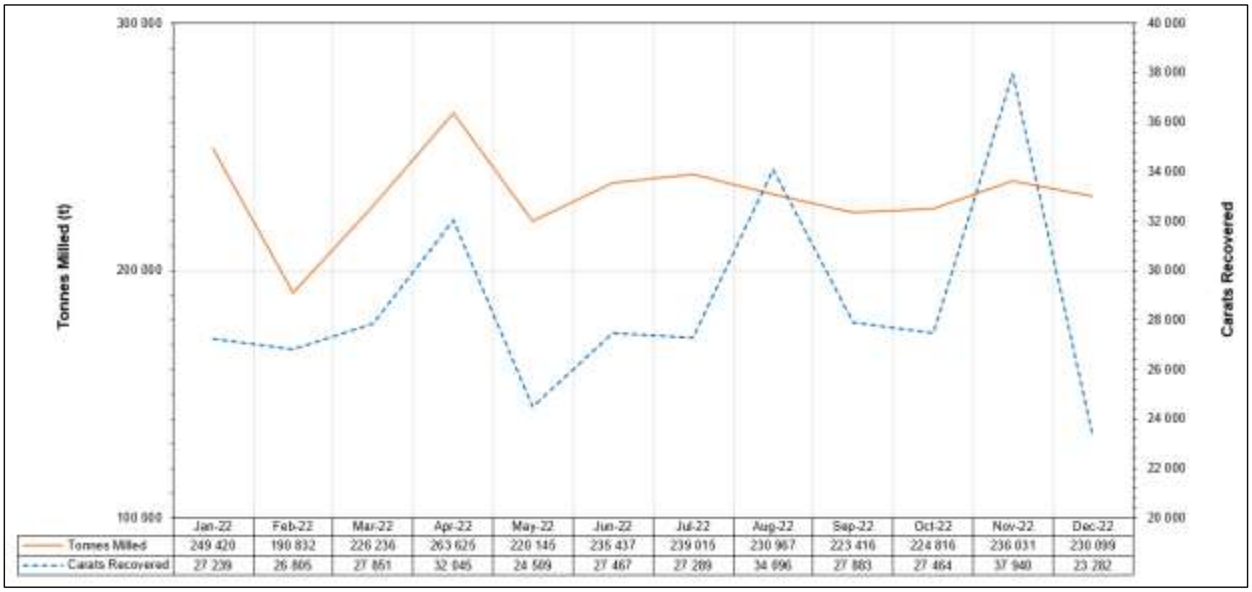
Since the conclusion of all MDR and Phase III work at KDM (expansion phases concluded in 2017), the following main plant upgrades and initiatives have been noted during the 02-03 September ’19 site visit, following discussions with various technical and management representatives from KDM:

- Wet dust scrubbing situated at the Primary Crushing section. This specific unit was commissioned during the December shutdown period in 2018;

- A Secondary Gyratory Crushing Feed Bin was installed as a separate optimization project by KDM and Lazenby in December ’18 with the following noticeable observations made:
  - A vibrator (mechanism) installed on the side of the bin discharge plating to assist with potential “bridging” due to possible slabby material received/encountered from the Primary Jaw Crushing section (function of the type of ore being fed as well as ore reduction amenability of the Primary Jaw Crusher based on ore type feed). Excess fines (predominantly weathered ore material) presented with ROM ore have also exacerbated the “bridging” issue historically; and
  - An operational bypass flexibility option exists regarding two vibrating feeders post new Secondary Gyratory Crushing Feed Bin arrangement: the Secondary Gyratory Crusher can be bypassed when associated downtime is experienced, or in case of excessive fines fed through the system (not purposely directed to the Secondary Gyratory Crusher).
- Wet dust scrubbing situated at the Pebble Crushing section. This particular unit was installed during the course of 2016. Subsequent to installation, the unit was repositioned and commissioned in August 2018;
- A Mill Relining Machine was procured after the Phase II expansion project was concluded in 2017;
- XRT Replacement/Refurbishment initiative conducted in 2020, replacement of 5 X-ray sorters due to corrosion;
- Phase II Audit XRT machine now utilized and incorporated as part of the mainstream plant in a primary “scavenger” application/duty;
- DMS/XRT Floats (i.e., Coarse Ore Stockpile) initiative: treatment of red area tailings with DebTech machines;
- Dust Suppression System Re-Starting initiative. The existing Dust Suppression System has been re-started at the end of August 2019 using Reverse Osmosis (R/O) Plant filtered water quality to combat ore transfer point dust emissions;
- Current R/O Plant capacity was expanded during 2021 to produce more R/O and/or filtered water quality quantities (volumes) for subsequent use in the treatment plant (regarding designated areas and associated users);
- A Recovery Plant Red Area Tails Dump treatment initiative for all associated stockpiles (inclusive of all Tertiary Crusher bypassed feed material also) active in mid-2023; and
- XRT Sorthouse upgrade was completed on 03 December 2018. Holding bins, feeders, washer driers and sort boxes were installed as part of the overall project. The main aim of the XRT Sorthouse upgrade project was to improve on washing and drying the concentrate product for increased (manual sorting) visibility. A secondary aim was also to eliminate sorter personnel returning heavy trolleys to the Recovery Plant to dispose of tailings. Currently, tailings material is being re-introduced back into the circuit via the Large Diamond Recovery (LDR) (XRT) Tailings Conveyor.

The following two graphs summarize 2022 plant performance in terms of milled tonnage, carats recovered and key treatment plant feed stream Particle Size Distribution (PSD) data.

Figure 17-6: 2022 Crushed/Milled Tonnage vs. Carat Recovery

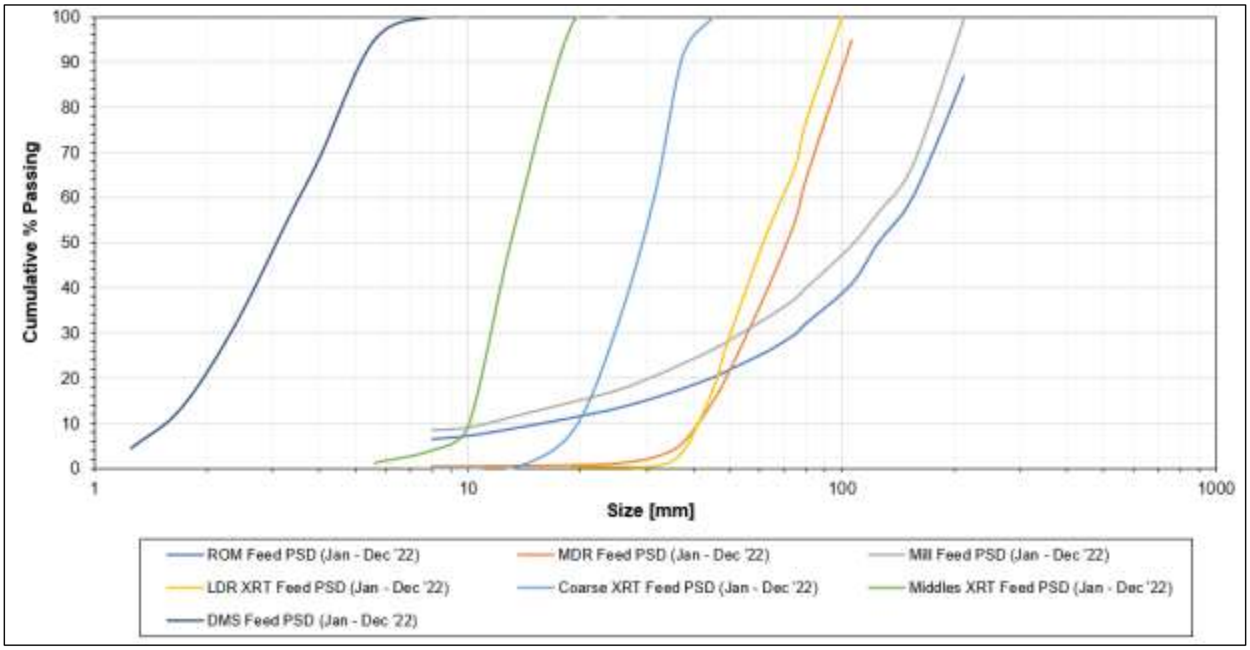


	Jan-22	Feb-22	Mar-22	Apr-22	May-22	Jun-22	Jul-22	Aug-22	Sep-22	Oct-22	Nov-22	Dec-22
Tonnes Milled	249 420	190 832	226 236	263 625	220 146	235 437	239 015	230 967	223 416	224 816	236 031	230 099
Carats Recovered	27 239	26 805	27 851	32 045	24 509	27 467	27 289	34 096	27 883	27 464	37 940	23 282

Source: Lucara (2023)

From above graph, lower monthly production was observed for February, May and September 2022 months respectively. The decreased production during these months was attributed to planned maintenance events.

Figure 17-7: 2022 Treatment Plant Key Feed Stream PSDs



Source: Lucara (2023)

Table 17-4 and Table 17-5 represent existing treatment plant panel aperture and crusher closed side setting (CSS) parameters.

Table 17-4: Key Screen Panel Aperture Summary

Screen Description	Screen Panel Aperture Size
MDR Screen (Double Deck)	Top: 100 mm SQ Bottom: 35 mm SQ
Bleed Screen (Single Deck)	40 mm SQ
Mill Discharge Screen (Double Deck)	Top: 100 mm SQ Bottom: 1.25 x 8.8 mm SLOT
Bulk Sorter Sizing Screen (Double Deck)	Top: 12 mm SQ Bottom: 7 mm SQ
XRT Sizing Screen (Double Deck)	Top: 40 mm SQ Bottom: 27 x 14 mm SLOT
XRT Tails Screen (Single Deck)	25 mm SQ

Screen Description	Screen Panel Aperture Size
DMS Dewatering Screen (Single Deck)	1.25 x 8.8 mm SLOT

Notes:
"SQ" denotes square aperture and "SLOT" denotes slotted aperture.
Source: Lucara (2019)

Table 17-5: Crusher CSS Summary

Crusher Description	Closed Side Setting (CSS) Size (mm)
Primary Jaw Crusher	180
Secondary Gyratory Crusher	65 – 90
Pebble Crusher	35 – 38
Tertiary Wet Flush Crusher	14

Source: Lucara (2019)

During 2018, DRA conducted a desktop evaluation for Lucara Botswana regarding front end plant modifications when considering UG mining and an associated capacity review if nominal ROM throughput should be increased to 3M tonnes per annum (Mtpa). The following key points summarizes some of the outcomes emanating from the desktop evaluation:

- It was envisaged that material would still be dumped in a primary tip, but it was to be modified to deal with excessive "mud" as there is anticipated water retention in the broken ground collapsing as part to of the sub level cave;
- Steel rebar from sub level cave support tunnels potentially entering the crusher;
- High carbon-containing materials with the potential of possibly presenting challenges:
  - In XRT – particularly the finer sizes which may survive the mill and have become higher purity, and the coarse fraction in the MDR which had not been ground down. Coal/carbon might report to the concentrate of XRT if of sufficient purity;
  - Thickening – keeping overflow clarity to a sufficient and desirable target whilst disposing of the lighter density carbon-containing material (with an affinity to float) to the TSF;
  - Acid mine drainage (AMD); and
  - Related to the possibility of spontaneous combustion of carbonaceous material on the "dead areas" of the plant (mill) feed stockpile.
- High waste contents diluting grade and requiring a tonnage increase to maintain diamond carat recovery, with options of waste sorting to possibly compensate for this. Part of the issue

identified here was the accuracy and integrity of waste grindability data, with the data indicating it was approaching the hardness of the host rock.

Another aspect that has been identified (apart from the desktop study completed in 2018) when considering UG mining and operations will be that of water management and potential impact(s) on the overall macro water balance when finding water at depth.

17.4 Process Plant Description

17.4.1 Crushing

Previous mill simulations and associated mass balances indicated that to achieve a head feed rate of 350-500 t/h processing hard (Unit 13/ M/PK(S)) ore, a secondary crushing stage is required ahead of the mill. The secondary crushing section stabilizes and reduces the mill load as well as the pebble crusher load. It also assists with top size feed control to the downstream milling section.

ROM material is delivered to the ROM tip by means of Articulated Dump Truck (ADT) and 777 rigid trucks and first stage crushing in the form of a Primary Jaw Crusher reduced ore to an acceptable feed envelope size ahead of the Secondary Crusher section.

Depending on the material treated, a proportion or the entire primary crushed ROM stream is diverted and processed through the secondary crusher circuit. Feed to the secondary crusher is scalped off the undersize on the MDR screen while the oversize removed on the same screen is partially sent to the crusher depending on a diverter setting; in addition, a portion (all or none) of the MDR tails can be sent to the Secondary Crusher. The Secondary Crusher product is reintroduced onto the Mill Stockpile Feed Conveyor with the screen undersize and bypass stream.

The +80 mm mill screen product and the 32 x 80 mm LDR XRT tailings are processed through the existing pebble crusher. The pebble crusher product is sized at 32 mm with all the +32 mm material reporting to the mill feed conveyor. A proportion of the -32 mm material bypasses the mill with the split balance of the -32 mm Bleed Screen undersize reporting directly to the mill feed conveyor. The bleed is required and balanced operationally to reduce mill loading.

The 20 x 32 mm tailings from the XRT Bulk Sorters are processed through a wet flush tertiary crusher circuit to liberate diamonds in this particular size fraction. The tertiary crusher product is reintroduced back into the circuit via Bulk Sorter Sizing Screen and reports to the relevant downstream process based on the crushed product size envelope.

17.4.2 Comminution – Milling, Bleed Screening and Pebble Crushing

Fresh mill feed is introduced into the mill from the feed stockpile along with a variable portion of the pebble crusher product directly. A bleed screen has been installed on the pebble crusher product stream, so that a proportion of the – 32 mm pebble crusher product can be bled out of the mill feed and report directly to downstream processes, thereby alleviating and balancing mill loading. The current AG Mill discharge grate incorporates Turbo Pulp Lifter technology to improve discharge and grate efficiency as well as withdrawal of material out of the mill.

17.4.3 XRT

The mill screen product (1.25 x 80 mm) is sized on the Bulk Sorter Sizing Screen and XRT Sizing Screen and the 32 x 80 mm oversize size fraction reports to the Large Diamond Recovery (LDR) XRT section. The purpose of the LDR is to recover large diamonds before the stream is processed through the pebble crusher circuit to reduce potential diamond breakage/damage. The 22 x 32 mm and 10 x 22 mm size fractions report to the Coarse and Middles Bulk Sorter sections respectively. The LDR XRT tailings is processed through the pebble crusher circuit. XRT tailings from the Coarse Bulk Sorters is transported to the Tertiary Crusher – passing over the XRT Tailings Screen first to separate the -20 mm. The combined Coarse and Middles Bulk Sorter tailings reports to the Scavenger (Audit) XRT and then have the option to be either diverted to the new XRT Audit Plant or be discarded as final coarse tailings on the DMS floats coarse ore stockpile.

17.4.4 DMS

As unweathered M/PK(S) and EM/PK(S) material is encountered from an ore treatment perspective, the denser the material becomes. High yields result in higher DMS cyclone sinks throughputs to the recovery circuit which can potentially become a bottleneck for the Recovery Plant. The existing Fines DMS plant processes the 6 x 1.25 mm size fraction and beneficiates diamondiferous concentrate from less heavy reject/gangue material. The Fines DMS throughput has been de-rated to accommodate the shift in current feed size treatment.

17.4.5 Recovery

The existing Recovery Plant processes the 5-6 x 1.25 mm size fraction received from the DMS section. In order to accommodate higher yielding M/PK(S) and EM/PK(S) material, a bulk reduction stage using MagRolls were initially added and incorporated as part of the original design. Since 05 September 2019 however, the MagRolls have been de-commissioned due to the conversion of the DMS plant from coarse to fines treatment (i.e., seeing less throughput) and due to the very low prevalence of magnetic diamonds observed in the DMS sinks yield portion ultimately reporting to the Recovery Plant. Other noticeable equipment located inside the Recovery consists of wet X-ray Machines, Infrared (IR) Drier and a dry Reconcentration X-ray Machine.

17.4.6 DMS Residue and Effluent Disposal

DMS tails, together with XRT tails and Degrit Screen grits are discarded as final coarse tailings on the DMS floats coarse ore stockpile.

All effluent streams generated in the plant (-1.25 mm) are pumped to the Degrit Effluent Cyclones situated at the Thickener. Overflow from the cyclones gravitates to the thickener feed well where flocculant at the correct solution strength is introduced to agglomerate and consolidate ultrafines for final disposal/removal to the Tailings Storage Facility (TSF) via Tailings Disposal Pump train. Underflow from the cyclones report to the Degrit Screen for fines dewatering and disposal to the DMS floats coarse ore stockpile.

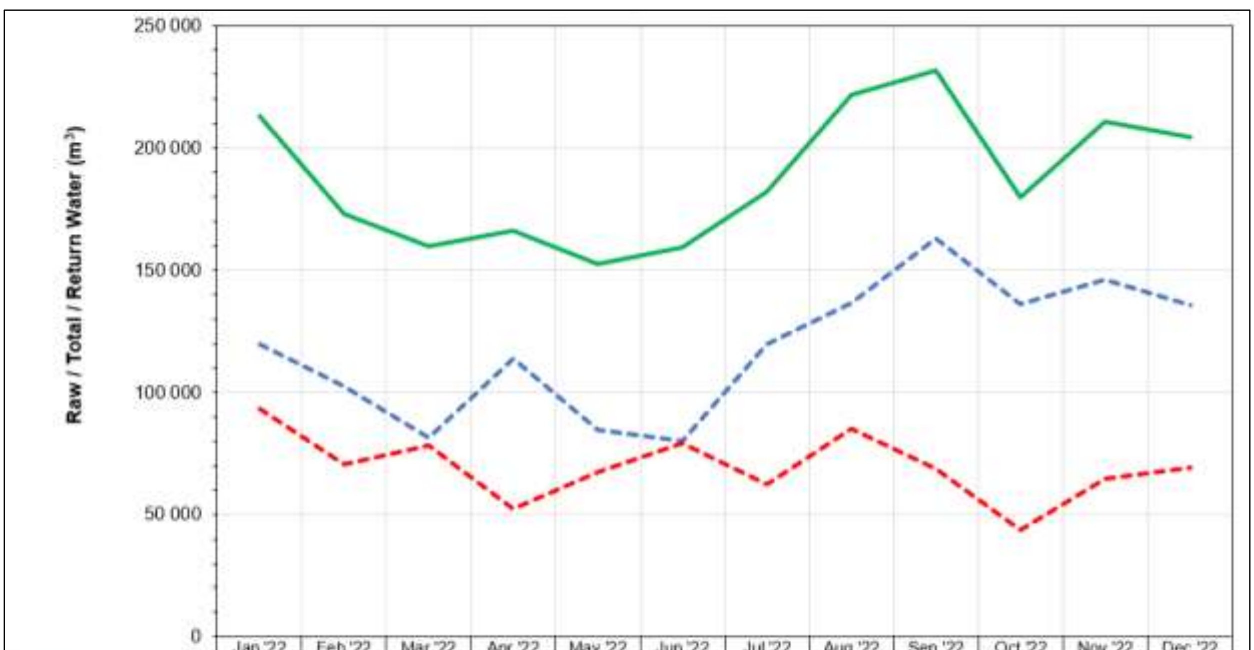
17.4.7 Services

Current plant-wide services at KDM's process treatment plant include instrument and process air from the respective compressors for valve actuation and XRT air-blow. Process water is collected and recycled back into the plant via Thickener and Process Water Tank. Raw Water is supplied to various end-users requiring borehole quality water for conversion to R/O, potable or filtered water quality via existing (and newly expanded) R/O Plant. Water chillers in the XRT and Recovery sections continuously cool down equipment. Dust suppression is utilized to combat dust emissions in especially the dry front-end section of the treatment plant.

17.4.8 Water Consumption

Water consumption data reported for 2022 is graphically presented in Figure 17-8 below. Raw water to the process treatment plant is supplied from pit dewatering and wellfields sources.

Figure 17-8: 2022 KDM Raw/Total Water Consumption



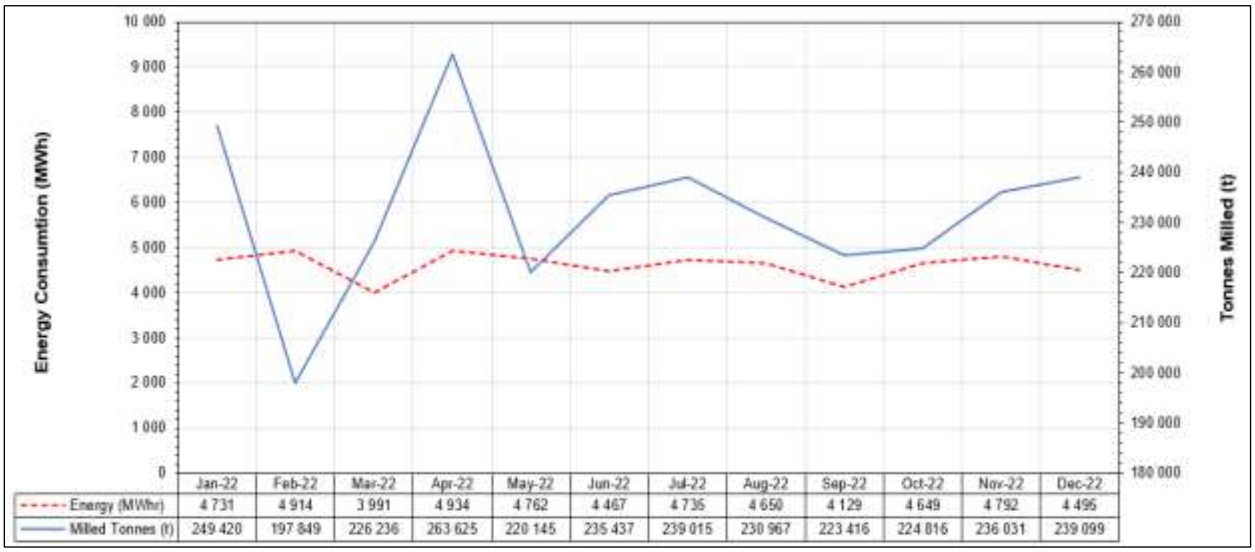
	Jan '22	Feb '22	Mar '22	Apr '22	May '22	Jun '22	Jul '22	Aug '22	Sep '22	Oct '22	Nov '22	Dec '22
Raw Water (m3)	119 739	102 473	81 434	113 869	84 823	80 121	119 715	136 528	162 803	136 323	146 092	135 482
Total Water (m3)	212 934	173 231	159 705	166 044	152 361	159 460	182 208	221 811	231 648	179 942	210 902	204 526
Return Water (m3)	93 195	70 758	78 271	52 175	67 538	79 339	62 493	85 283	68 845	43 619	64 810	69 044

Source: Lucara (2023)

17.4.9 Energy Consumption

Energy consumption data (associated with the process treatment plant) observed for the 2022 period is reported and summarized in Figure 17-9 below.

Figure 17-9: 2022 KDM Energy Consumption



Source: Lucara (2023)

18 PROJECT INFRASTRUCTURE AND SERVICES

18.1 General Site Arrangement

The existing operating mine and processing facility have a fully established services and infrastructure base built up from the mine's 10 years of full-time operation.

The UGP has taken advantage of many of the existing facilities on site including:

- The processing plant;
- Site access road;
- Airstrip;
- Pit dewatering pipeline;
- Maintenance facilities;
- FRD facilities;
- Waste rock dump;
- Coarse reject facility;
- Explosive magazines;
- Ancillary mobile equipment; and
- Bulk fuel storage.

Since 2020 The UGP has added a number of Project-specific items including but not limited to the following:

- New power supply for both the existing OP mine and processing plant and the UGP:
  - A new 220/132 kV substation and 132 kV switchyard at Botswana Power Corporation's 400/220 kV Letlhakane substation;
  - 29 km-long, 132 kV overhead powerline from the BPC Letlhakane substation to the KDM substation; and
  - 132/11 kV substation and switchyard located at KDM.
- Distribution of 11 kV power from the KDM substation to the Project site;

- UGP pad surface substation, e-house and power distribution;
- Eight MW of diesel generator back-up power;
- Reverse-osmosis plant capacity increase for UG water supply;
- Sewage treatment plant upgrades to handle the UGP including the camp;
- Phase 1 (two paddocks) of a new FRD;
- Modern 200-person capacity camp complex to support the construction workforce;
- Internal infrastructure pads and roadways;
- Surface sediment pond for managing UG dewatering;
- UG pad buildings and facilities to support the operation including:
  - UGP office complex;
  - Change house for UG personnel;
  - Maintenance shops;
  - Warehouses;
  - Chemical grout mixing;
  - Lamp room;
  - Line out rooms;
  - Training and meeting rooms; and
  - Local first aid room.

Major surface facilities remaining to be built for the UGP include the main mine exhaust fans, UG bulk air coolers, the permanent P/S personnel and material winder and saline water management evaporators and containment pond.

The general site layout of the UG pad is shown in Figure 18-1.

Figure 18-1: UG Pad Area with Existing Buildings and Infrastructure

18.2 Water Management

As presented in Section 16.4, hydrogeological modelling forecasts large volumes of saline water with an estimated total dissolved solids (TDS) of ±30,000 mg/L (approximately the same as seawater) will be pumped to the surface during the UG mine life once development begins within granitic basement rock located some 500 m below surface. Treating this water by Reverse Osmosis (RO) is both costly and technically prohibitive in addition to exceeding the quantity and quality requirements of the site water consumers. The water pumped from the UG mine will, in lieu of RO treatment, be disposed of by means of mechanical evaporation. A fully lined surface pond will be constructed adjacent to the UG operations to receive the saline water from the UG mine and to pump it to modular mechanical evaporators which are designed to expedite the natural evaporation process. As the water evaporates it will leave in the basin of the pond the salts contained in UG mine water, eventually filling it. Pond expansions will be carried out during the mine life as needed to meet the demand of the UG water dewatering volumes and chemistry.

This section presents the management of the UG saline water that will be pumped to the surface continuously over the mine life.

18.2.1 Volumes and Chemical Composition of the UG Saline Water

Table 18-1 gives the estimated flow rates of saline water to be pumped to the surface by the future UG mine. The flow rates in Table 18-1 are taken from Itasca updated 3-D Hydrogeological Model September 21, 2023 (refer to Section 16.4). Up to 380 m^3/h (peak flow) are planned to be pumped to the surface in 2027, then slowly decreasing over time and to stabilize at an average daily flow rate of 250 m^3/h at year 2036 up to the end of the mine life.

Table 18-1 also provides the salt loads of the UG saline water with an estimated TDS concentration of circa 30,000 mg/L. Salt loads are based on the estimated monthly average daily flow rates pumped to the surface in a given year, rather than being calculated on the basis of peak flow rates which would result in overestimating the salt loads.

Table 18-1: Estimated Volumes of UGP Saline Water 2026-2040

Year	Estimated UG Groundwater Inflows Requiring Disposal*		Salt Management
	Estimated Average Hourly Flow (m^3/h)	Estimated Average Daily Flow (m^3/d)	Salt Load in UG Saline Water @ 30,000 mg/L TDS (t/yr dry mass)
2026	153	3,679	40,285
2027	353	8,483	92,889
2028	355	8,510	93,200
2029	327	7,840	85,900
2030	310	7,450	81,500
2031	296	7,110	77,800

Year	Estimated UG Groundwater Inflows Requiring Disposal*		Salt Management
	Estimated Average Hourly Flow (m^3/h)	Estimated Average Daily Flow (m^3/d)	Salt Load in UG Saline Water @ 30,000 mg/L TDS (t/yr dry mass)
2032	285	6,830	74,800
2033	275	6,600	72,300
2034	267	6,420	70,300
2035	260	6,250	68,400
2036	255	6,120	67,000
2037	250	6,010	65,800
2038	246	5,910	64,700
2039	243	5,820	63,800
2040	240	5,750	63,000
Total/Avg	**278 average**	**6,660 average**	**948,500**

*Source: Itasca Updated 3-D Hydrogeological Model September 21, 2023– Refer to Section 16.4

The chemical composition of the UG saline water was studied as part of the KDM Water Management Project in 2022/2023. Samples of deep water were collected from two (2) deep boreholes (Exigo1 BH and Vent Shaft BH). Laboratory analysis of these samples showed TDS concentrations in the range of 25,000 to 33,000 mg/L. The dissolved solids in the water are predominantly sodium chloride, calcium and sulfates; all other constituents of the water are present at very low or trace concentrations. The key results of the lab analyses are presented in Table 18-2. The lab work also included full metals scan which did not reveal any significative concentrations for other constituents of the UG water. It is noted in Table 18-2 that the UG water temperature (measured in the field at the time of sampling) was consistently at 40º C which is an indicator of water coming from the deep lithologic formations.

Table 18-2: Chemical Composition of the UG Saline Water

Date/ Location	pH	Temp ºC	TDS mg/L	Ca mg/L	Mg mg/L	Na mg/L	Cl mg/L	SO4 mg/L	K mg/L	Fe mg/L	Mn mg/L
07/04/22 Field	7.48	39ºC	26,900	2848	170	7428	17995	286	47	0.17	0.13
18/03/22 Lab ID Wellfield VS BH	7.69	Not reported	25,074	3776	34	7000	17995	594	32	1.0	0.09

Date/ Location	pH	Temp ºC	TDS mg/L	Ca mg/L	Mg mg/L	Na mg/L	Cl mg/L	SO4 mg/L	K mg/L	Fe mg/L	Mn mg/L
13/07/22 Lab ID Wellfield VS BH	7.56	39 ºC	27,846	3168	131	8000	18370	595	42	1.6	0.67
28/07/22 Lab ID Wellfield VS BH	7.36	40ºC	32,266	2968	301	3333	11000	622	30	0.95	0.42
28/07/22 Lab ID Wellfield Exigo1	7.29	39ºC	31,764	2496	389	3333	10000	521	29	0.48	0.38
29/08/22 Lab ID Wellfield Exigo1	7.48	40ºC	31,255	-----	-----	-----	13696	610	-----	-----	----
29/08/22 Lab ID Wellfield Exigo1	7.54	40ºC	31,120	-----	-----	-----	14497	544	-----	-----	-----
30/08/22 Lab ID Wellfield Exigo1	7.47	39ºC	32,125				13996	596			
30/08/22 Lab ID Wellfield Exigo1	7.63	39ºC	30,705	-----	----	-----	13796	559	-----	-----	----
20/09/22 Lab ID Wellfield VS BH	7.39	Not reported	28,832	----	---	----	16146	722	---	----	----
21/09/22 Lab ID M&L SA VS BH	8.25	Not reported	33,048	2826	36	6651	18895	1006	43	0.64	0.25
4/10/22 Third Party Lab UGP BH	7.4	39ºC	30,464	3726	32	6234			48	<0.02	0.3

Date/ Location	pH	Temp ºC	TDS mg/L	Ca mg/L	Mg mg/L	Na mg/L	Cl mg/L	SO4 mg/L	K mg/L	Fe mg/L	Mn mg/L
07/11/22 Lab Wellfield Exigo1	6.6	Not Reported	28,224	----	----	----	26893	863	----	----	----
07/11/22 Lab Wellfield UGP BH	7.1	Not Reported	27,972	-----	-----	----	18895	863	----	-----	-----
30/11/22 Lab Wellfield Exigo1	7.6	Not Reported	24,759	3500	72.9	9000	18000	665	32	0.14	0.16
02/12/22 Exigo1	7.7	Not Reported	24,696	3560	58.4	8000	17620	417	35	0.08	0.6

18.2.2 Overview of the Site Water Circuit

A key principle in the management of the volumes of UG saline water is that none of this water can and will be used by the process plant which cannot tolerate such high levels (refer to the above Table 18-2) of TDS in its operation. This is illustrated by Figure 18-1 showing that the management of the UG saline water will be done outside of the process plant water circuit. In terms of water, the only link between the process plant and the UG mine is filtered and RO water supply to the mine as shown by Figure 18-1.

The KDM process plant water circuit in Figure 18-1 is a quasi-closed loop. All major water lines at the process plant are equipped with flowmeters that report continuous readings to the control room and which then feed the process plant daily water balance.

Aside from stormwater collected by the different water storage infrastructures at the process plant (Process Dam, Overflow Dam, Slimes Storage Facilities, Storm Water Pond), the only external source of water to the process plant is the dewatering and in-pit boreholes that achieve the depressurization of the OP. The dewatering of the OP brings to the process plant an average of 220 m^3/h of water (24-hour averaged hourly pumping rates; the 220 m^3/h average is based on 25 months of on-line measured flow rates). This water from OP dewatering is fed to the process plant (refer to Figure 18-1) together with water recovered from the slimes storage facility (SSF) to sustain its water balance by compensating for the following losses of water:

- Water captured by the deposited slimes in the SSF and including seepage losses from the facility;
- Water captured by the coarse tailings;
- Domestic uses of water and potable water; and

- Water lost by evaporation from the different water storage infrastructures at the process plant.

Under the constant supply of the OP dewatering water (average 220 m^3/h), and in spite of the water losses listed above, the process plant water balance remains positive (excess of water). The steady state average daily excess of water from the process plant is 70 m^3/h (24-hour averaged hourly flow rates continuously measured) and this excess is sent to the neighboring Orapa mine under an agreement between Lucara and Debswana.

<u>Salt Balance Within the Process Plant Water Circuit.</u>

The OP dewatering water sent to the process plant is at an average TDS concentration of 3,000 mg/L. Due to the continuous reuse (via the thickener) of the salty return water from the SSF (average 100 m^3/h refer to Figure 18-1), and due to solar evaporation from the large water storage infrastructures, a build-up of salts takes place within the process plant water circuit which is controlled by the following bleeds of TDS:

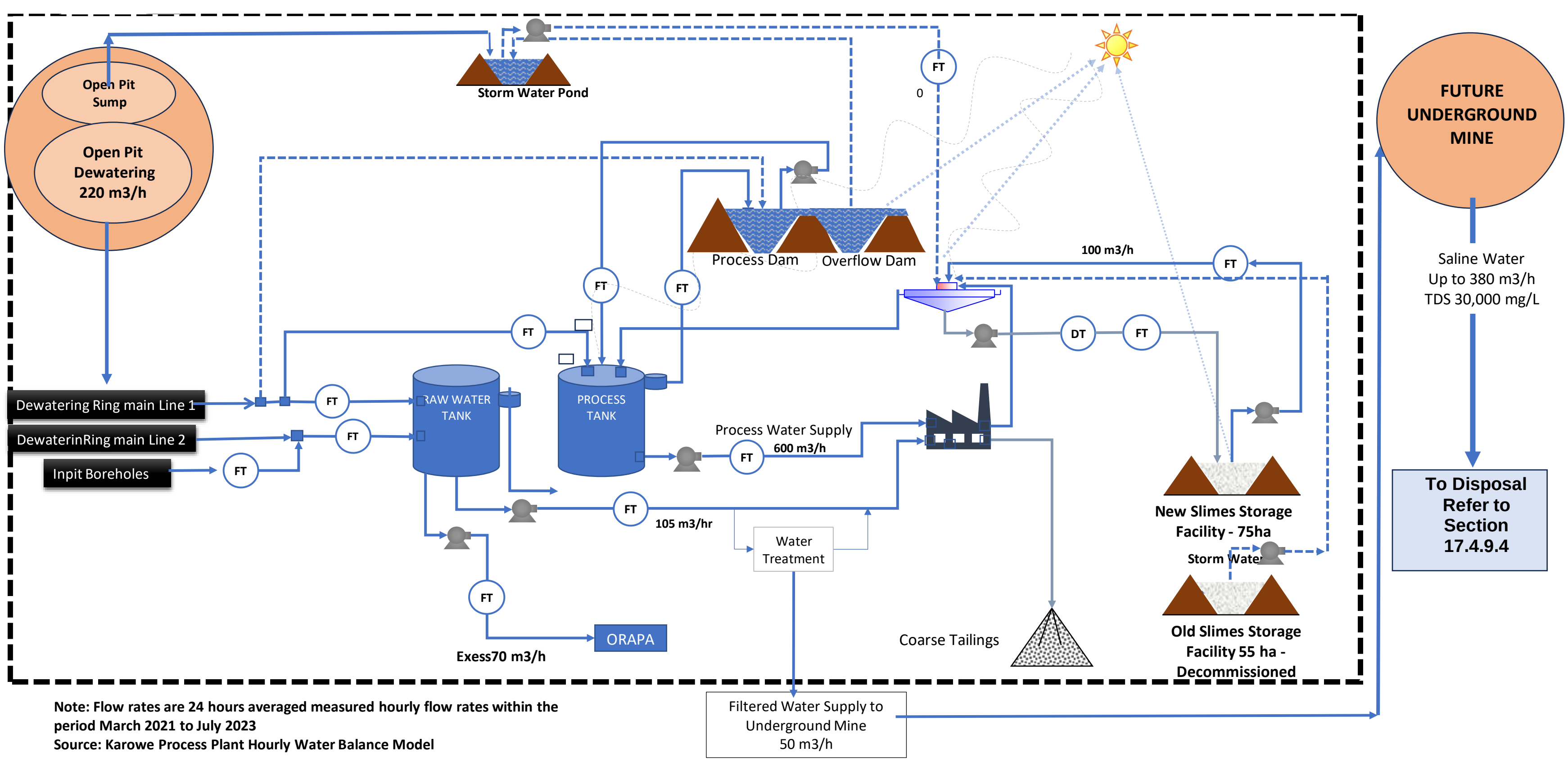
1) TDS bleed via the excess water sent to Orapa;

2) TDS retained with the pore water in the SSF; and

3) TDS retained by the coarse tailings.

A regular monitoring of the TDS concentrations is performed in all main water lines of the water circuit to ensure the salts balance is maintained below 4,000 mg/L TDS.

Figure 18-2: Site Water Circuit



Note: Flow rates are 24 hours averaged measured hourly flow rates within the period March 2021 to July 2023
Source: Karowe Process Plant Hourly Water Balance Model

18.2.3 Water Supply to the Process Plant Over the Mine Life

As indicated in Section 18.2.2 above, the source of water supply to the process plant is the OP dewatering water, currently 220 m^3/h.

Itasca updated 3-D Hydrogeological model (refer to Section 16.4) shows that the OP dewatering flows to the process plant will decrease over time (2024 – 2040) from 220 m^3/h (current), down to about 165 m^3/h in 2026, and further down to 125 m^3/h by 2030 and the subsequent years.

Such a reduction in the volumes of dewatering water will not, however, result in a risk for the process plant to face a water supply deficit during the mine life. Running the process plant water balance model under dewatering flows lower than the125 m^3/h expected in 2030 shows the water balance to remain positive and to continue generating an excess water to Orapa. Furthermore, during the rainy season (November to March), significant volumes of stormwater will be collected by the new SSF (75 ha) and will contribute to maintain a positive (excess) water balance at the process plant.

18.2.4 Disposal of the UG Saline Water

Referring to Table 18-1, considerable volumes of saline water will have to be disposed of over the mine life. Surface discharge at the mine site of such considerable volumes (up to 350 m^3/h) of water at some 30,000 mg/L TDS is unacceptable within best water management practices, Botswana permitting, flooding of mine site areas, infiltration and salinization of groundwater upper aquifer used by farmers.

As part of the KDM Water Management Project in 2022/2023, four (4) options were studied extensively, including cost estimating, for the disposal of the UGP saline water. These options are summarized in Table 18-3 along with the key criteria for the selection of the disposal option.

Table 18-3: Disposal Options for the UGP Saline Water

Description of the Options	Criteria
1) Pumping of the saline water to a natural salt pan	• Compatibility of the UGP saline water with the natural environment of a salt pan • Relatively low capital and operating costs • Simplicity of operation • Not feasible in terms of permitting
2) Reverse Osmosis treatment and return to the community of the treated water	• Capital intensive under a design flow rate of 350 m^3/h • Very high O&M costs over 15 years of operation • No design flexibility to adapt to uncertainties with the volumes of UGP saline water predicted by modelling – risk of exceeding design flow and shut down of treatment operations • No design and operational flexibility to uncertainties with the TDS concentrations in the UG saline water

Description of the Options	Criteria
	• No design flexibility to handle stormwater that will report to the UG mine • Ponds required for the disposal of brine rejects from RO and storage of salts
3) Solar Evaporation Ponds combined with CSR opportunity to harvest and sell salts by local community	• Considerable land requirements for solar evaporation calculated to + 100 ha • Work with the BotAsh salts production facility in Botswana confirmed that the salts contained in KDM UG water are not marketable
4) Mechanical Evaporators	• Proven technology under dry arid climatic conditions as those at KDM • Operational flexible capacity to both, variations in volumes of UG saline water and to TDS concentrations • Operational flexibility to dispose of stormwater reporting to the UG mine • Option with the lowest capital cost • Lowest O&M costs • Maintenance simplicity • One single pond required for the operation of the mechanical evaporators and storage of salts

Based on the assessment that is summarized in Table 18-3, mechanical evaporation was selected as the most suitable, flexible and adaptive option for the disposal of the UGP saline water at KDM under the operation of the future UG mine.

18.2.5 Application of Mechanical Evaporators Technology for the Disposal of UG Saline Water

Mechanical evaporation of excess water at mine sites (tailings facility, OP dewatering, brine from RO treatment) is a proven technology under dry arid climatic conditions as those at KDM and is used in dozens of other applications regionally and throughout the world.

The application of Mechanical Evaporation requires the construction of a pond where the mechanical evaporators will be installed and whose functions are:

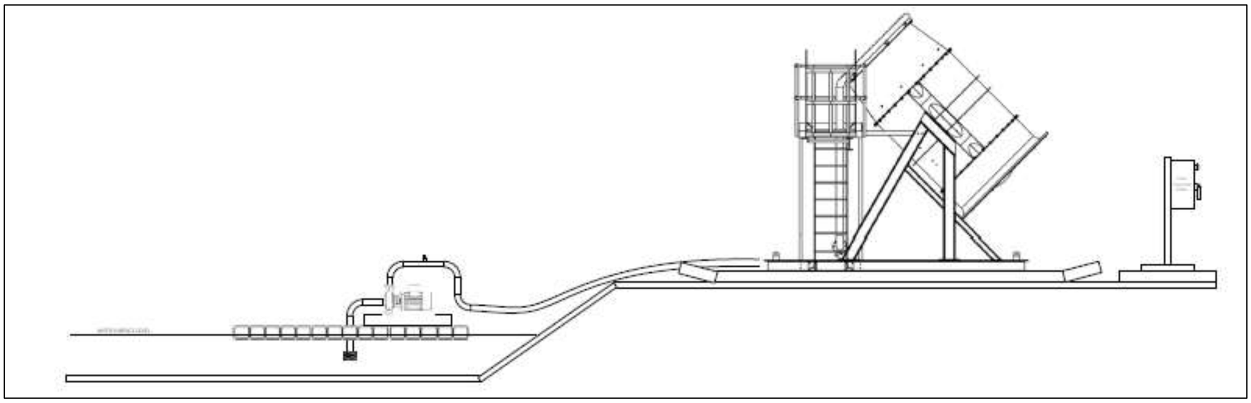
1) to receive the saline water pumped from the UG mine;

2) to feed the mechanical evaporators; and

3) to provide storage for the salts.

This is illustrated by the diagram in Figure 18-3.

Figure 18-3: Conceptual Arrangement of Mechanical Evaporators and Pond



The application of the mechanical evaporators will first be implemented to dispose of the saline water that will be pumped during the five (5) first years of the UG mine development (refer to Table 18-4), namely for years 2026 to 2030. After the first five (5) years, mechanical evaporation will continue by expanding the pond for additional storage of salts as required during the course of the mine operation; however, the future expansion will be based on revised, actual design and operating parameters the first five years of operation will have demonstrated. The final pond configuration will be designed taking into account actual flow rates pumped from UG and the actual TDS concentrations in this water.

The implementation for the first five (5) years will be based on the following design:

- Installation of twelve (12) mechanical evaporators, 75 kW each, capable of achieving the evaporation of the daily volume of water to be received by the pond. The operation plan of the mechanical evaporators for the first five (5) years is summarized in Table 18-4. From this table, it is seen how the operation of the mechanical evaporators can adapt to the decreasing average daily flow rates by managing the number of the daily evaporative hours (operating hours); and

- Construction of a pond sized for a volume of 570,000 m^3, excluding freeboard, and with a footprint of 7.4 ha. The sizing of the pond is governed by the quantity of salts to be stored in the pond from the evaporation operations, and by the depth of water (1.5 m) to be maintained in the pond at all times for the operation of the mechanical evaporators.

Table 18-4: Operation Plan of the Mechanical Evaporators First 5 Years

Year	Avg Flow Rate from the UG Mine to the Evaporators (m^3/h)	Average Operating Hours per day per Evaporator (h/d/evaporator)	Salts Storage (t/y)
2026	153	5	40,300
2027	353	10	92,900
2028	355	10	93,200
2029	327	9	85,900
2030	310	8	81,500

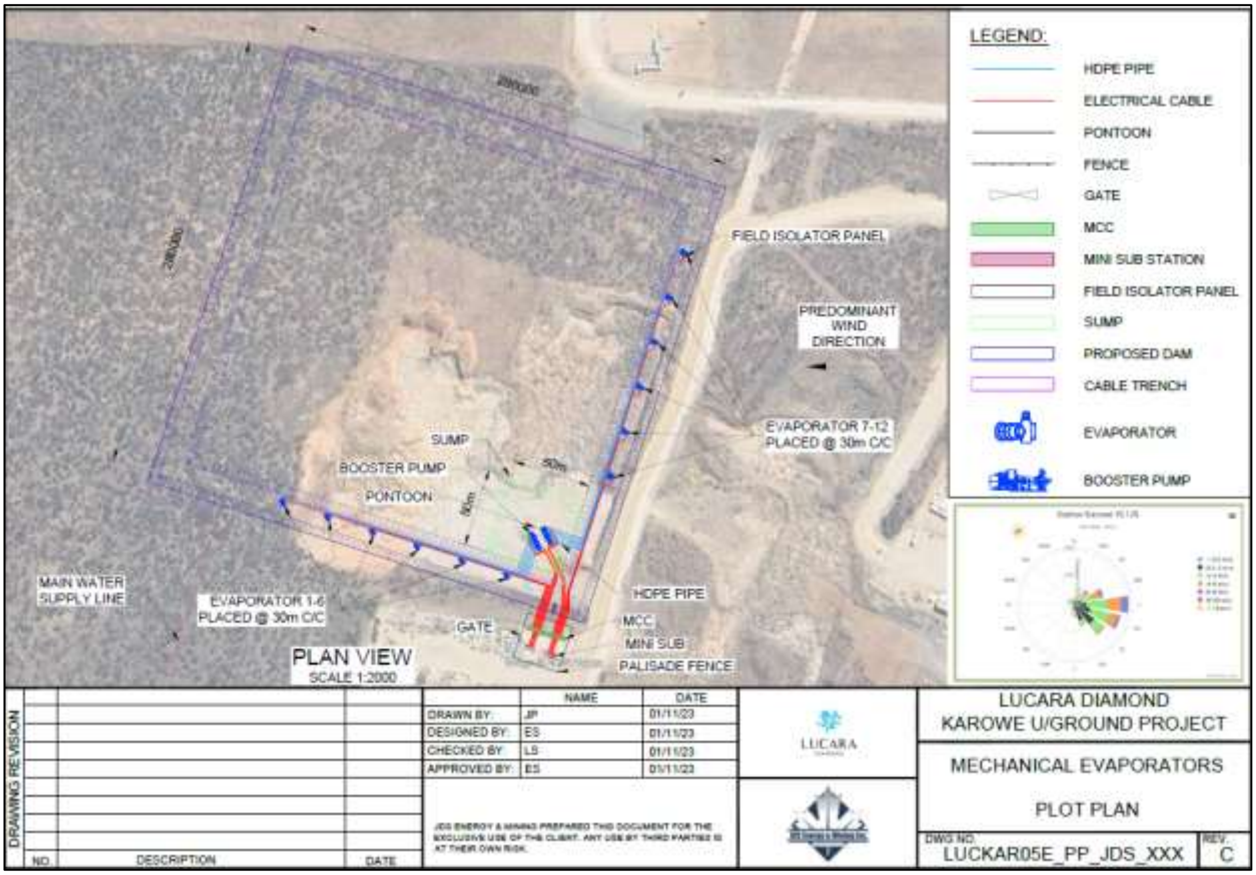
Figure 18-4 shows the siting of the pond, located at some 1 km of the shaft area. The pond will be constructed at the location of an abandoned borrow area. The general arrangement (GA) of the pond in Figure 18-4 is for the first five (5) years of the UG mine development and operation. It shows the footprint of the pond, 7.4 ha in area, along with the positioning of the 12 mechanical evaporators. The positioning of the evaporators takes into account the following key design factors:

1) The directions and velocities of the dominant winds; and

2) The distribution within the pond of the residual salts solution and of the crystalized salts that is left after evaporation of the UG mine water.

The piping that will take the saline water from the UG to the surface will be extended along the pond up to its upper edge in Figure 18-4. From there, the saline water will be distributed over the upper edge of the pond and will be extracted at the other extremity of the pond to feed the evaporators.

Figure 18-4: General Arrangement of Pond and Mechanical Evaporators – First Five (5) Years 2026 - 2030



As the saline water received by the pond is evaporated, a saturated solution of salts will progressively build up at the bottom of the pond, followed by progressive crystallization of the salts and accumulation at the bottom of the pond over the five (5) years of operation. After the first five (5) years of evaporation operations (2026-2030), and depending on the actual volumes of salts accumulated, the pond will be expanded to pursue the disposal of the UG saline water during mining.

Closure to the pond will be part of the overall closure plan of KDM.

18.2.6 Management of Storm Water Reporting to the Future UG Mine After Breaking of the Crown Pillar at Year 2034

For the period 2026 to 2034, the storm water collected by the OP will not report to the UG Mine; it will be pumped from the OP sump to the storm water pond as done currently and shown by Figure 18-2.

In 2034, it is planned to wreck the crown pillar between the UG and OP, thus creating a hydraulic connection between the two. At this time, the storm water will report to the UG and is planned to be pumped to the surface for disposal together with the groundwater inflows to the mechanical evaporators.

The design basis for the management of stormwater is the 1:100 4-day rain intensity, which is aligned with the design basis for the new SSF that was commissioned in 2023. Section 16.4 – Mine Dewatering flow rates for the volumes of stormwater to be managed by the UG mine dewatering system.

18.3 Waste Rock Management

Waste rock from the OP and UG development is hauled to a large waste rock facility located on the west side of the mine property. The dump has more than enough capacity to hold the remainder of OP and UG planned waste. The dump has a number of survey monitoring points strategically positioned to monitor movement or settling over time, neither of which has been an issue. All of the host waste rock at KDM is benign from an acid-rock drainage/metal leaching point of view so no special treatment is done for the waste rock other than following designs for geotechnical stability purposes.

18.4 Residue (Tailings) Storage Facilities

18.4.1 Introduction

A feasibility study for the design of FRD 1 was carried out in 2019 by Knight Piésold (KP) consulting. The study proposed increasing the height of FRD 1 to a final elevation of 1042 masl and a new FRD 2 abutting FRD 1, (previously labelled as Phase 1 and Phase 2 respectively), with Phase 2 (FRD 2) being built to an elevation of 1042 masl (Knight Piesold, 2019).

Lucara Botswana has adopted GISTM as best practice for fine and coarse residue disposal and as a result KP was appointed to re-design the Phase 2 FRD in 2021 with height restrictions being imposed on FRD 1 to the current lift elevation of 1031 masl. The area for Phase 2 FRD will store tailings up to the end of 2025. The site was further constrained as the 2019 Phase 2 design site extended further south beyond the existing fence; however, the design has now been limited to fall within the fence area. The final 2021 design, which began construction of starter walls in 2022 is approximately 1300 m wide and 500 m long, with a divider wall creating two paddocks which were subsequently labelled 2A and 2B respectively (Knight Piésold, 2022).

The FRD 2 was designed to be lifted in two stages. Stage 1, the starter wall was constructed with local borrow and the Stage 2 lift will be constructed using waste rock. The final design height was approximately 10 m, up to elevation 1026 masl. A site selection study was undertaken in 2022 for FRD options that can accommodate the Life of Mine (LOM) tailings, and a new site on the West of the existing FRD’s (FRD 1 and 2) was selected by the client (Knight Piésold, 2022).

This new site has been labelled FRD 3 for reference and the detailed design of this Facility commenced in 2023.

The design criteria for FRD 2 and 3 also limits the final elevation to 1031 masl (approximately 15 m in height above NGL). Deposition into FRD 2 is currently underway as per the planned OP production up to the year 2025 (Knight Piésold, 2022).

Contained within this section is the 2023 feasibility designs for the mine residue storage facilities referred to as the Coarse Residue Deposit (CRD), FRD 2 and FRD 3.

18.4.2 Design Criteria

The design criteria in Table 18-5 represents the requirements as of June 2023. Additional geotechnical and hydrogeological test works are being undertaken on site to ascertain the foundation conditions of the facility, prior to the completion of the full design. A Design Basis Memo has been produced identifying the approved final footprint that shall be undertaken for the life of the mine.

Table 18-5: Design Basis for Modelling CRD Extension and the Development of FRD 3

Criteria	Units	Design	Source
Life of Facility Required	Yrs.	15 (2026 to 2041)	Lucara Projections
Run of Mine (ROM)	Mt	37.0	Lucara Projections
Tonnages of CRD and FRD Generated	Percentage Split	40% to FRD and 60% of Tonnes to CRD	Karowe Diamond Mine
Coarse Residue Deposits (CRD)			
Tonnes to Facility	Mt	22.2	Lucara Projections
Density	t/m³	1.7	(Royal HaskoningDHV, 2017)
Volume for Life	Mm³	13.05	Lucara – Survey (2023)
Maximum Height	m	36	Lucara – Survey (2023)
Design Slope	Ratio (V:H)	1:1.5	Lucara – Survey (2023)
FRD – FRD 2 and 3 (2026 – 2041)			
Tonnes to Facility	Mt	14.8	Lucara Projection
Volume for Life	Mm³	13.5	Lucara Projections
Maximum Crest Elevation	masl	1031	(Knight Piesold , 2023)
Impoundment Wall Inside Slope	Ratio (V:H)	1:2	(Knight Piesold , 2023)
Impoundment Wall Outside Slope	Ratio (V:H)	1:3	(Knight Piesold , 2023)
Crest Width	m	10	(Knight Piesold , 2023)
Estimated Area Required	Ha	156	(Knight Piesold , 2023)

Criteria	Units	Design	Source
Minimum freeboard		Minimum of 0.8 m above normal operating FRD pond level plus inflow from 1:50 year 24 hr. rainfall event.	(*National Water Act* 36, 1999)
Consequence Classification (CCS)		High (FRD 2 at 1026 m)	(Knight Piésold, 2022)

Source: KP (2023)

In order to model the footprint of the CRD, the planned production data between 2023 to 2026 was taken into consideration using survey data which represents site conditions as of December 2022. Production tonnages adopted for this study are provided in Table 18-6 and Table 18-7 and reflect the mining and processing plan for this study.

Table 18-6: Production Tonnages for OP Mining

Year	Annual Tonnages (Mt)		
	CRD	FRD	TOTAL
2023	1.7	1.13	2.83
2024	1.7	1.13	2.83
2025	0.20	0.14	0.34
2026	0.90	0.60	1.50
		Total	**7.51**

Source: KP (2023)

Table 18-7: Production Tonnages for UG Mining

Year	Annual Tonnages (Mt)		
	CRD	FRD	TOTAL
2026	0.06	0.04	0.1
2027	0.66	0.44	1.1
2025	1.62	1.08	2.70
2026	1.62	1.08	2.70
1027	1.62	1.08	2.70
2028	1.62	1.08	2.70
2029	1.62	1.08	2.70

Year	Annual Tonnages (Mt)		
	CRD	FRD	TOTAL
2030	1.62	1.08	2.70
2031	1.62	1.08	2.70
2032	1.62	1.08	2.70
2033	1.62	1.08	2.70
2034	1.62	1.08	2.70
2035	1.62	1.08	2.70
2036	1.62	1.08	2.70
2037	1.62	1.08	2.70
2038	1.62	1.08	2.70
2039	1.62	1.08	2.70
2040	1.62	1.08	2.70
2041	0.06	0.04	0.10
		Total	**36.4**

Source: KP (2023)

18.4.3 Geotechnical Investigation

Geotechnical investigations were undertaken to assess the properties of the material in the footprint area of FRD 2, and whether there is suitable material for the expansion of FRD 2 (Knight Piésold, 2022). Medium dense to lose aeolian sand covers the FRD site, this soil compacts well but will not create an impervious barrier. No suitably impervious starter wall materials were located during the investigation. From the investigation it was concluded that modified soils such as a bentonite mixture can be considered for impervious embankment zones if required.

The design for FRD 2 was completed to be raised to final height of 1026 masl. The current Phase 3 designs and geotechnical investigations are being undertaken to identify wall building material borrow pits and select material for the construction of FRD 3 and the raising of FRD 2 to the elevation of 1031 for LOM design.

Aeolian sand and powdered calcrete are susceptible to surface water erosion and dispersion, they should not be used for wall construction (Knight Piésold, 2022), the recommended materials for starter wall construction are nodular calcrete (G5 to G7 quality) and calcareous silty sand (G7 quality).

The embankment layout for FRD 2 and FRD 3 are expected to be similar in design, with the key difference being that FRD 3's embankment crest being constructed at a width of 10 m, instead of 6 m, due to the long lengths of the wall and to improve traffic management on the crest of the facility. The availability of sufficient construction material will be confirmed at the end of the current geotechnical investigations which are expected to be completed before the end of

February 2024. A key consideration required is to ensure that the haul distances are minimized, and further work will be required closer to the start of the construction phase of FRD 3, that the material for wall building is secured up front, prior to the engagement of a contractor.

18.4.4 Coarse Residue Deposit (CRD)

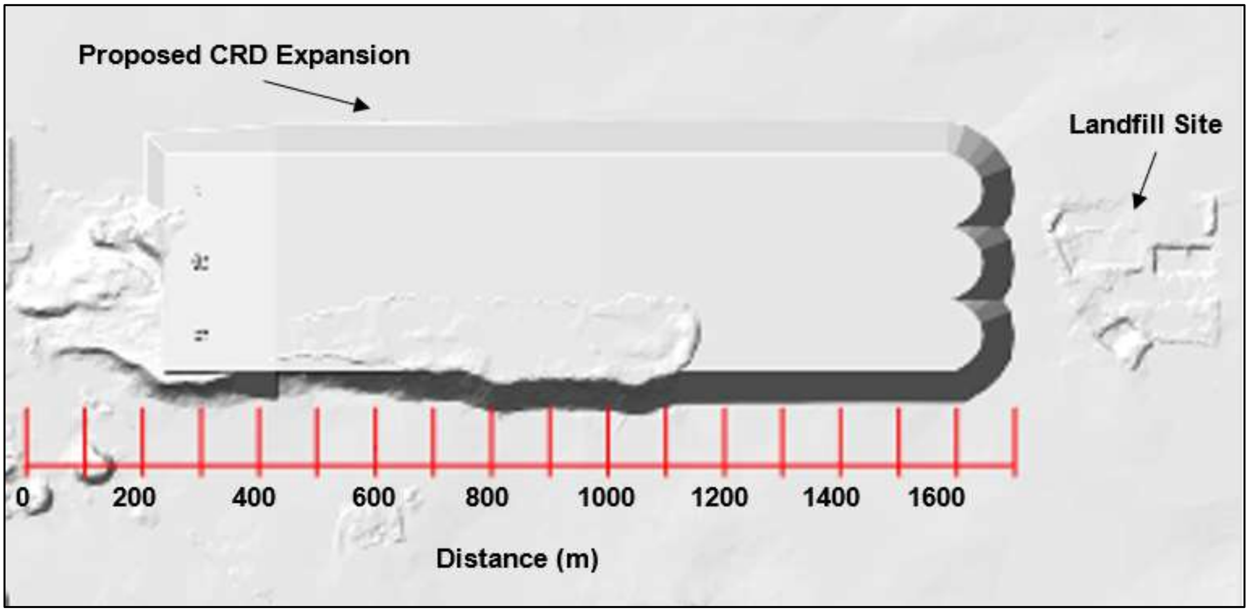
The coarse residue material will be deposited at the current CRD facility which will expand south to accommodate LOM production. A capacity analysis was undertaken to quantify the required expansion of the CRD using tonnage projections provided by Lucara.

The existing CRD facility utilizes a single conveyor system, which builds the CRD from North to South and a bypass conveyor that builds from East to West and is on standby when the conveyor system is being extended. As of 2022, the bypass conveyor had built up sufficient material for the client to start a new conveyor 125 m towards the East of the first conveyor line. For ease of reference, the original conveyor that ran from North to South, has been labelled as Leg 1. Leg 2 depicts the current location where KDM is depositing coarse tailings material.

In order to optimize the storage to space ratio as well as redundancy, it is proposed that a three-leg conveyor system be implemented. The Leg 1 conveyor, according to the December 2022 survey, was terminated at approximately 1200 m from the start of the conveyor system. The existing conveyor system would need to continue depositing on its current leg in a southerly direction until it reaches the final position as indicated on Figure 18-5, which is roughly 1650 m from the start of the conveyor system.

Further modelling will be required (to be reviewed on a 3-year basis), in order to plan the deposition to the end of life as the final length is subject to change. This is due to variability in the density of material placed and the moisture content of the material. Elevation differences between different legs should be carefully managed to avoid pooling and damming of water on top of the CRD. Excess water or water storage on the CRD will impact the stability of the facility.

Figure 18-5: 3D Model Developed Using MUK3D Software



Source: KP (2023)

In addition to accommodating tonnages to 2041, the proposed layout of the CRD will remain within the mine lease boundary and will not encroach on the landfill sites. This needs to be frequently monitored as the CRD development progresses. Taking the above into consideration, a CRD design was developed, this is summarized in Table 18-8.

Table 18-8: Summary of Proposed CRD Facility Design Characteristics

Parameter	Units	Value
Design Life (2023 – 2041)	Years	2026-2041
Total Storage Required (2023 – 2041)	Mm^3	13.05
Total Storage Achieved	Mm^3	15
Crest Elevation	m	1052
Maximum Height of Facility	masl	36
Number of Conveyor Legs	No.	3
Distance Between Conveyors (centre to centre)	m	125
Assumed Side Slope of Facility	Ratio (V:H)	1: 1.5
Distance from toe of CRD to Mine Lease Area	m	Approx. 270
Distance from Landfill Sites from CRD toe on the southern extremity	m	Approx. 112

Source: KP (2023)

18.4.5 Fine Residue Deposit

18.4.5.1 FRD 1

With FRD 1 limited to a maximum elevation of 1031 masl, the facility has been filled to maximum capacity. Closure plans for the facility are currently underway.

18.4.5.2 FRD 2

The current deposition method on site for the FRD 2 facility is to place the material upstream of the impoundment wall using the spigotting method. The facility is divided into two paddocks, and the impoundment walls are raised in phases to ensure there is sufficient capacity for fine residue deposition. The spigots are opened and closed to control the flow of the slimes into the basin at any given time. The deposition plan ensures that deposition controls pond location, creates beach freeboard and allows for adequate drying/ consolidation of the slimes. The water is pumped from the facility directly back to the plant.

With the FRD 1 facility at its maximum capacity, the current design for the FRD 2 facility allows for deposition and residue storage until the end of the year 2025. The required expansion of the FRD 2 facility to LoM is restricted as follows:

1) To the south of the FRD 2 facility, expansion is limited by the mine lease boundary;

2) To the east, the site landfill and future CRD footprint;

3) The existing FRD 1 facility to the north; and

4) Topsoil dump on the western side of the facility.

In addition to the location constraints identified above, the ratio of waste rock to slimes storage was determined to provide a relative comparison of cost and overall design efficiency. This ratio was used as a comparison factor between the options in determining which option would be further developed. The options considered for selection were for the proposed FRD 3 facility and the possible further expansion of the existing FRD 2 facility (limited to 1031 masl). To evaluate which option would be best, a Multi-Criteria Analysis (MCA) was completed to determine which FRD option would be further developed.

Thirteen options were evaluated, and Option 12 was selected as the preferred option. Option 12 did not cross the zone of relaxation and it accommodates the tonnage profile. It was found that less facilities or services must be relocated for Option 12 compared to other options, hence it was the most practicable option (Knight Piésold, 2022).

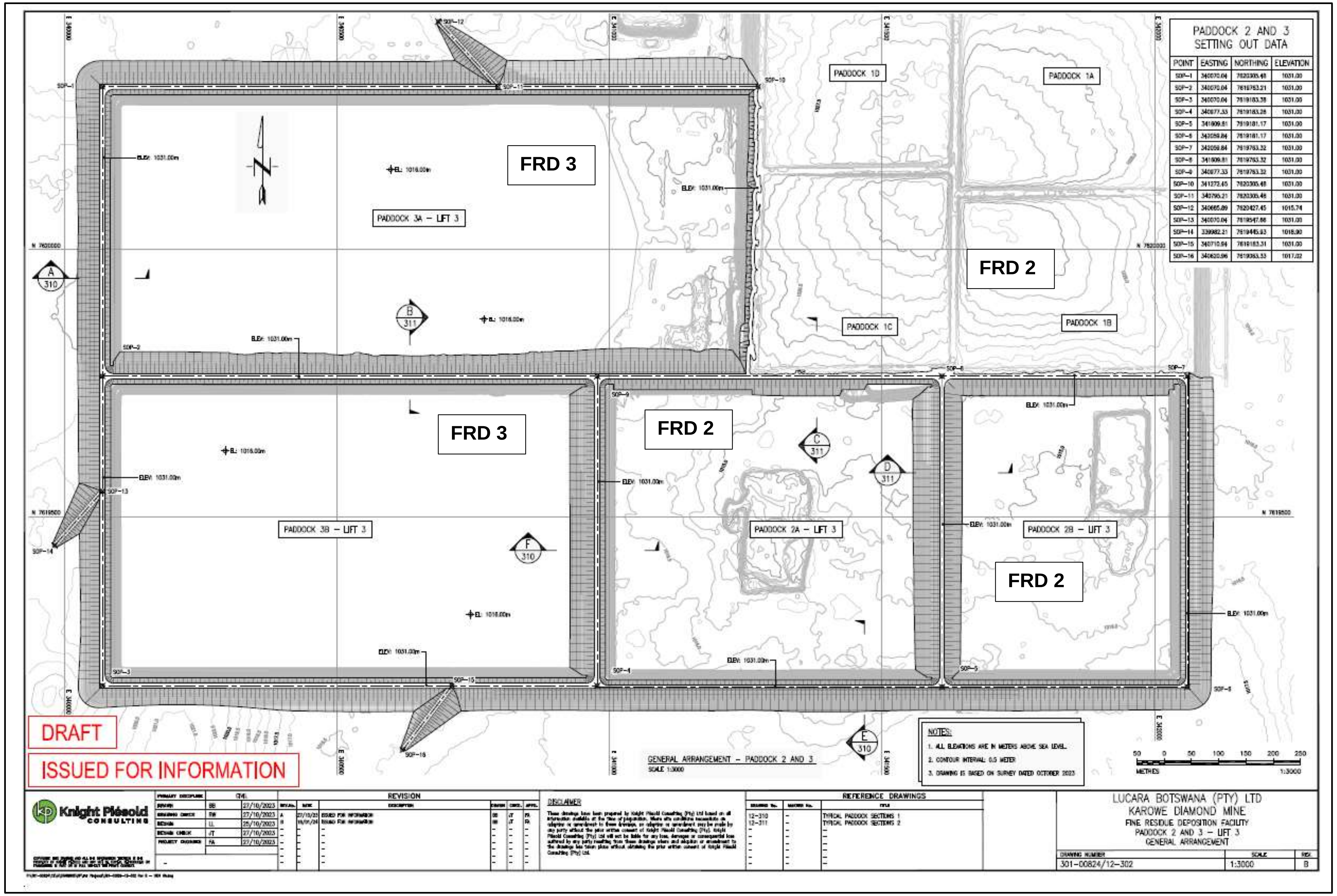
During the optimization of Option 12 the footprint was reduced slightly to reduce the overall impact on the environment, and to improve cost efficiency by adjoining FRD 1 and 2 to FRD 3, so that a new eastern wall does not need to be constructed, refer to Table 18-9 for further details. In addition, the optimization study showed potential cost savings, by raising FRD 2 to the elevation of 1031 masl as well (from the initial 2026 masl design level), bringing all the facilities to the same height. This will also assist with overall operational efficiency and reduces the chances of cascading dam failures from one facility to the other. Figure 18-6 shows the general layout of the proposed FRD facilities.

Table 18-9: Summary of Proposed FRD 3 Design Characteristics (Knight Piesold , 2023)

Description	Units	Value
Design Life from 2026	Years	15
Total Storage Required from January 2026	Mm^3	13.6
Total Storage Achieved from January 2026	Mm^3	15.1
Crest Elevation	masl	1031
Height of Facility	m	15
Volume of Material Required	Mm^3	5.8

Source: KP (2023)

Figure 18-6: FRD 2 and FRD 3 - Proposed Layout Drawing



Source: KP (2023)

The proposed design of the FRD 2 and FRD 3 resulted in facilities with the following features:

<u>FRD 2</u>

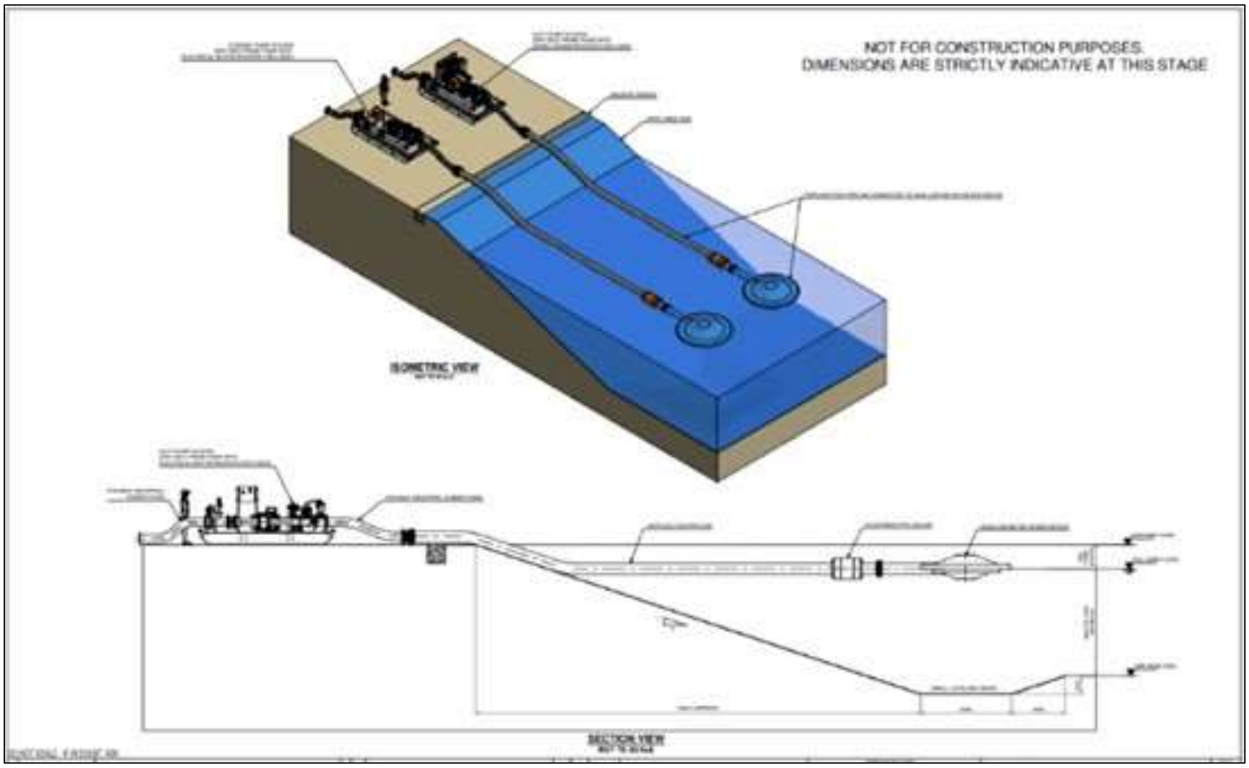
- The impoundment wall will be raised to a final height of 1031 masl from the previous design elevation of 1026 masl (Knight Piésold, 2022);
- The elevation adjustment shall be achieved through three successive lifts: a 2-m lift (referred to as “lift 1.2”), a 5-m lift (referred to as “lift 2”), and a final 5-m lift (referred to as “lift 3”). The wall elevation shall be increased by adding 0.5 m layers of selected waste rock, which should be compacted according to the developed site-specific standard;
- The impoundment walls are designed with a crest width of 6 m; and
- The downstream slope of the walls shall have a ratio of 1 vertical unit to 3 horizontal units (1V:3H), while the upstream slope will have a ratio of 1 vertical unit to 2 horizontal units (1V:2H).

<u>FRD 3</u>

- The impoundment wall will be raised to a height of 1031 masl. The starter wall shall be raised using 500 mm thick layers of selected calcrete, compacted to the site-specific developed standard;
- The FRD is planned to be divided into two paddocks for better pool control;
- The impoundment walls are designed to have a 10 m wide crest to provide easier access and traffic control on top of the facility; and
- The downstream and upstream slopes of the embankment walls are set at 1V:3H and 1V:2H respectively.

The FRDs accumulated process water, together with any stormwater shall be removed through a decant system comprising of pump units connected to the turret system as shown on Figure 18-7. The decant water from the dam shall be conveyed back to the plant. The stormwater accumulated during a storm event will be conveyed to a new stormwater dam to be located between the landfill and FRD 2.

Figure 18-7: Pump Unit and Turret System Proposed for the FRD 3 Facility



Source: KP (2023)

18.4.6 FRD Capacity Analysis

The final storage capacities for FRD 2 and FRD 3 were determined using MUK3D modelling software. In addition to the storage capacities, the required time to deposit the calculated volumes was also determined. These results are presented in Table 18-10 and Table 18-11.

Table 18-10: FRD 2 Capacity Volumes and Time to Fill

Cell	Planned Berm Elevation (masl)	Height Above Natural Ground Level (m)	Volume achieved (from 2026) (m^3)	Months	Years	Total Years
Raise 1						
Paddock 2A	1019	3	1,262,20	15.6	1.3	2.1
Paddock 2B	1019	3	822,100	10.2	0.8	

Cell	Planned Berm Elevation (masl)	Height Above Natural Ground Level (m)	Volume achieved (from 2026) (m^3)	Months	Years	Total Years
Raise 2 (Current Lift)						
Paddock 2A	1021	2	475,20	5.9	0.5	0.9
Paddock 2B	1021	2	425,60	5.3	0.4	
Raise 3						
Paddock 2A	1026	5	1,671,600	20.7	1.7	2.9
Paddock 2B	1026	5	1,132,400	14.0	1.2	
Raise 4						
Paddock 2A	1031	5	1,428,800	17.7	1.5	2.5
Paddock 2B	1031	5	977,800	12.1	1.0	
		Total	**5,789,100**			**9**

Rate of rise = 1.7 m/year
Source: KP (2023)

Table 18-11: FRD 3 Capacity volumes and Time to Fill

Cell	Planned Berm Elevation (masl)	Height Above Natural Ground Level (m)	Volume Achieved (from 2026) (m^3)	Months	Years	Total Years
Raise 1						
Paddock 3A	1019	3	1,532,900	19.0	1.6	2.9
Paddock 3B	1019	3	1,246,500	15.4	1.3	
Raise 2						
Paddock 3A	1021	5	1,031,800	12.8	1.1	2.0
Paddock 3B	1021	5	868,900	10.8	0.9	
Raise 3						
Paddock 3A	1026	5	2,733,100	33.8	2.8	5.2
Paddock 3B	1026	5	2,301,900	28.5	2.4	
Raise 4						
Paddock 3A	1031	5	2,349,100	29.1	2.4	4.5
Paddock 3B	1031	5	1,979,000	24.5	2.0	
		Total	**14,043,200**			**15**

Rate of rise = 1.2 m/year
Source: KP (2023)

18.4.7 FRD 2 Classification

At the time of this report, no Tailings Dam Breach Analysis (TDBA) had been performed for FRD 2 and FRD 3 at 1031 masl.

Knight Piésold has performed a TDBA study for KDM FRD 2 for the 2025 design crest elevation of 1026 masl (Knight Piesold, 2022). A “High” consequence classification was recommended. The corresponding flood criteria from the Global Industry Standard for Tailings Management (GISTM) is for a storm with an annual exceedance probability of 1/2,475, presented on Table 18-12. This was applied as the flood design criteria for the development of rainy-day failure modes as part of this TDBA.

Table 18-12: Recommended GISTM Flood Design Criteria (Knight Piésold, 2022)

Consequence Classification	Flood Criteria – Annual Exceedance Probability (AEP)	
	Operations and Closure (Active care)	Passive-Closure (Passive care)
Low	1/200	1/10,000
Significant	1/1,000	1/10,000
High	**1/2,475**	**1/10,000**
Very High	1/5,000	1/10,000
Extreme	1/10,000	1/10,000

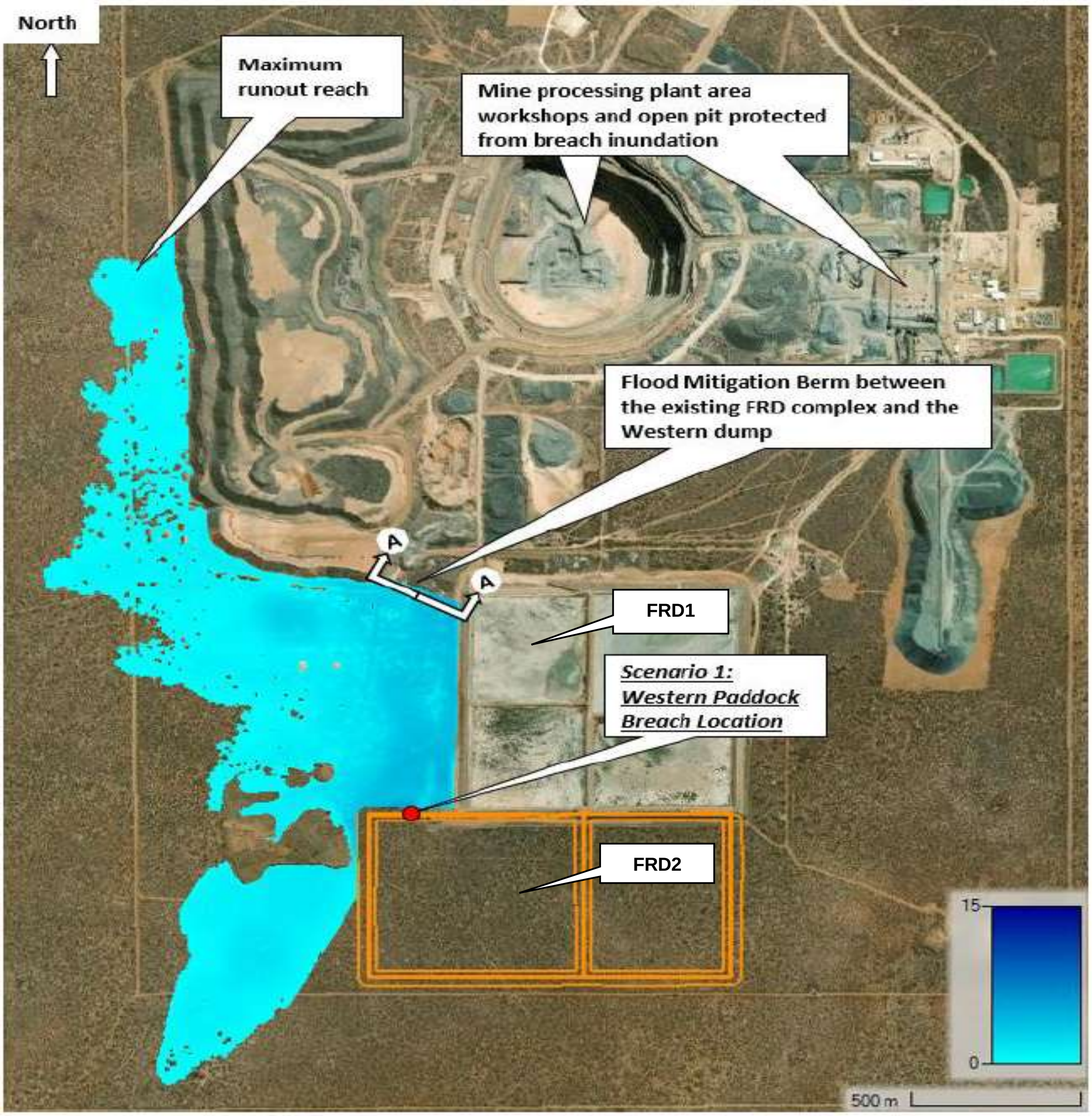
Source: KP (2023)

Two flood-induced failure scenarios were considered in the study, i.e., a failure at the highest embankment sections corresponding to the Western and Eastern paddocks of the FRD. A breach of the Western paddock was evaluated to be the most critical scenario in terms of both the Population at Risk (PAR) and Potential Loss of Life (PLL). A breach event at this position has the potential to inundate the area between the existing FRD and the pit, the mine processing plant, workshops, and admin buildings towards the northern site boundary. The pit itself would also be subject to flooding. Figure 18-8 shows the breach of the Western paddock which has an inundation area of 3.52 km² and a runout distance of 4.5 km.

A “High” Consequence Classification was recommended for the proposed FRD 2 in terms of the GISTM classification system, this increased classification is attributed to the identified PAR for a failure within the Western paddock of the proposed FRD and due to significant economic losses for the mine owners. In terms of the SANS 10286 classification, the FRD would be classified as “Medium hazard”.

The proposed construction of a flood mitigation berm in the area between the existing West Waste Rock dump and the existing FRD complex as shown on Figure 18-8, would deflect the breach flow from the Western paddock to the west side of the site around the existing waste dump. This would mitigate any flooding to the main mining processing areas and the centrally located OP. The construction of such a berm would lead to a lowered consequence classification i.e., from a “High” to “Significant” consequence classification.

Figure 18-8: FRD 2 Facility Western Breach Scenario (Knight Piésold, 2022)



Source: KP (2022)

18.4.8 Stability Assessment

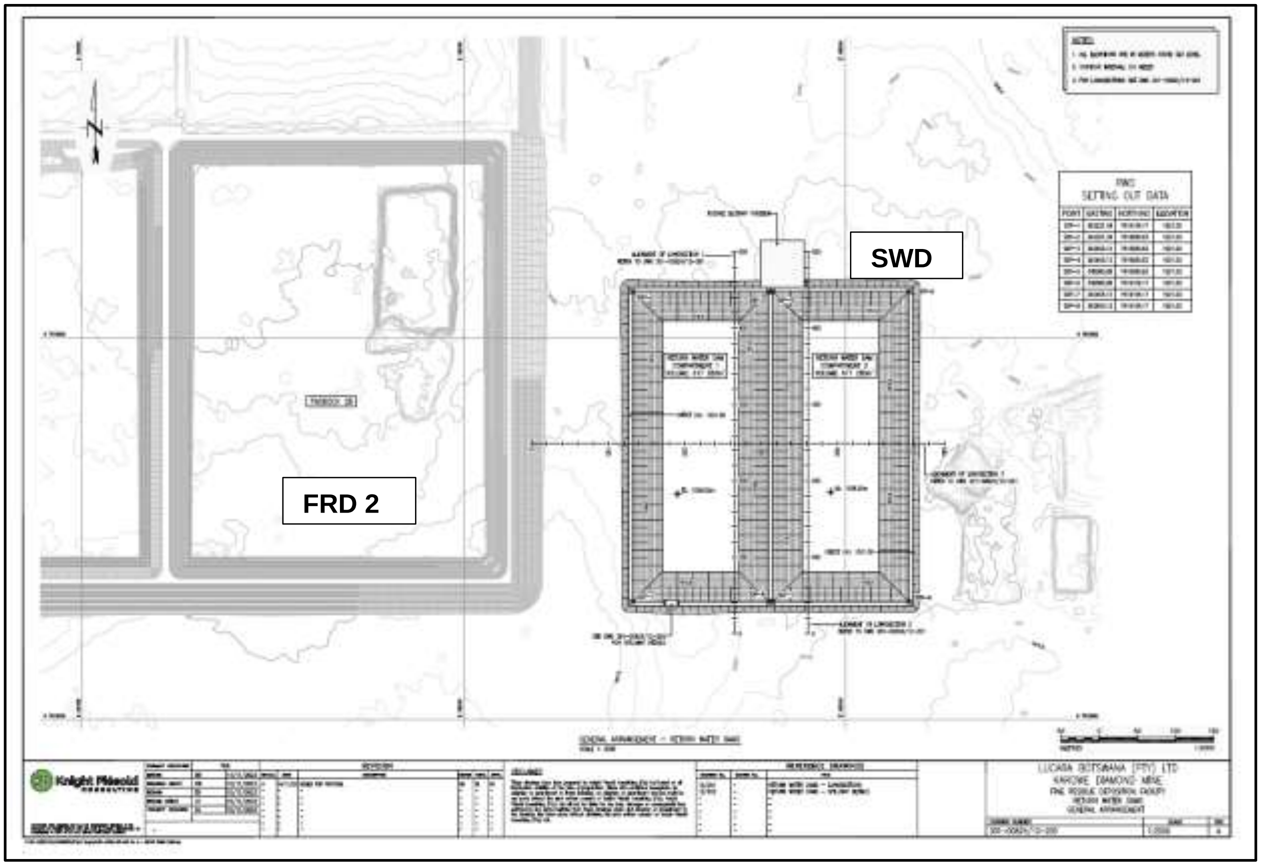
At the time of this report, the FRD 3 stability assessment had not been concluded for the final FRD heights but will be completed before April 2024.

18.4.9 Storm Water Management

A stormwater management plan for the FRDs was developed to mitigate the risk of the site becoming inoperable during major storm events, reduce any rain induced instability and to prevent contaminated runoff water from entering any water resource as per GN704. Typically, this involves diverting non-contact water (natural runoff upstream of a site that has not come into contact with mining-related surfaces) through a berm upstream of the site. The stormwater mitigation design includes the construction of the berm from suitable available material to a height of 1.5 m with side slopes of 1 in 1.5 and a crest width of 1 m (Knight Piésold, 2022).

It has been proposed that contact water (runoff within the site that has come into contact with mining-related surfaces) generated from the embankment walls of the FRDs be diverted away from the site perimeter using a stormwater channel (Knight Piésold, 2022). The stormwater channel is planned to be unlined earth canals that convey contact runoff along the perimeter of the FRDs, before discharging to a suitable location in the environment away from the FRDs. The channel will fall with the natural ground slope of the land to avoid excessive excavation, routing it to a controlled discharge point.

Figure 18-9: Proposed Storm Water Dam (SWD)



Source: KP (2023)

18.4.10 Water Balance

In order to demonstrate that the capacity of FRD 2 is sufficient in containing water generated from process water and direct precipitation, a daily time-step volumetric water balance was modelled (Knight Piésold, 2022). The water balance was modelled according to the proposed deposition strategy and considers losses due to entrainment, seepage, evaporation, and re-use. With FRD 1 being in care and maintenance, the deposition was cycled between the two paddocks of FRD 2, with only one paddock being actively deposited on at a time (Knight Piésold, 2022).

A design storm event of 1: 2,475, corresponding to the facility's high rating, was selected based on the GISTM. When stress tested, the model indicated that the FRD 2 facility would reach a minimum freeboard of 180 mm and 60 mm for paddocks A and B, respectively, under extreme storm conditions and no spill event (Knight Piésold, 2022).

Based on South African guideline GN 704, the minimum operating freeboard for a facility that stores contact water is 800 mm above the full supply level, which is achieved by both the paddocks. This freeboard is to be monitored by the mine operators under the guidance of the Engineer of Record (EoR).

A new Storm Water Dam (SWD) has been proposed with the expansion of the FRD 2 and a new FRD 3 facility. During storm events, excess water from these facilities shall be delivered to the SWD through reinforced concrete lined silt trap for settlement of suspended solids (Knight Piésold, 2023).

An HDPE geomembrane liner has been proposed to ensure containment and prevent water seepage from the RWD.

18.4.11 Conclusion

In summary, the FRD and CRD facilities can be expanded to accommodate the proposed UG mining extension and shall be designed to comply with GISTM and GN704 requirements.

19 MARKET STUDIES AND CONTRACTS

19.1 Lucara Diamond Marketing

This section is contributed by Lucara under the oversight of Dr. John Armstrong. The information documented herein is updated where relevant to end of June 2023.

Under the terms and conditions contained within ML 2008/6L, Boteti will hold open tenders for sale of diamonds in Botswana. In the period 2012 to the end of 2014, dual viewing of goods was 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. In 2020, during the Covid-19 pandemic Lucara received permission to hold dual viewings of tender goods in Gaborone and Antwerp with final tender closing in Antwerp, as of June 2023 this mechanism was still in effect. In Q1 2018, Lucara acquired Clara Diamond Solutions (Clara). Clara is Lucara's 100% owned proprietary, secure web-based digital marketplace which is best suited to transact diamonds between 1 and 15 ct, in better colours and quality. The Clara platform matches buyers to sellers on a stone-by-stone basis based on polished demand and is the only sales platform in the world that uses technology to provide complete assurance on diamond provenance. Clara continues to gain scale and interest as the financial benefits of purchasing rough diamonds in this innovative way are realized for all participants and, buyers become more focused on transparency and traceability of diamonds from mine to retail. A portion of the KDM production, mainly diamonds in the range from 6gr to 10.8ct in the better colours, shapes and clarity are sold through the Clara platform. Clara has a client base of >100 customers and sells diamonds on average every 4 to 5 weeks depending on supply.

KDM's large, high value diamonds have historically accounted for approximately 60% to 70% of Lucara's annual revenues. In 2020, Lucara announced a partnership agreement with HB Antwerp (HB), entering into a definitive sales agreement for diamonds recovered that exceed +10.8 ct from Lucara's 100% owned KDM in Botswana. This agreement was extended with certain amendments during 2021 and in November 2022, the agreement was extended again for a further ten-year period through December 31, 2032. Under the sales agreement, +10.8 ct gem and near gem diamonds from KDM of qualities that can directly enter the manufacturing stream are being sold to HB at prices based on the estimated polished outcome of each diamond. The estimated polished value is determined through state-of-the-art scanning and planning technology, with an adjusted amount payable on actual achieved polished sales, less a fee and the cost of manufacturing. If the final sales price is higher than the initial estimated polished price a true up payment is payable to Lucara. Any manufactured diamonds sold to an end buyer for less than the initial estimated polished price (after deductions for HB's fee and the cost of manufacturing) will result in the difference being refunded to HB. Top-up payments, net of manufacturing costs, are paid when polished diamonds are sold to an end buyer and the sales prices achieved exceed the initial purchase price paid to Lucara. Top-up payments primarily relate to carats delivered in previous quarters. The amount and timing of top up payments received is impacted by the complexity of certain rough diamonds and the qualitative assumptions that are part of the initial planning process. At various points during the manufacturing process, the stones are re-assessed, and adjustments may be made to the manufacturing plan, with the objective of maximizing the final sales price.

All +10.8 ct non-gem quality diamonds and all diamonds less than 10.8 ct in weight which did not meet the criteria for sale on Clara are being sold as rough through a quarterly tender. Lucara manages a rough price book (>4000 price points) that generates a reserve price for each sales lot sold via tender. The Government Diamond Valuator (GDV) also completes a valuation of the rough lots to be tendered, sold via Clara or the HB Antwerp Diamonds sales agreement and reserve prices are compared prior release. 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' sales through the Clara platform and final polished value sold under the HB Antwerp sales agreement.

19.2 Diamond Sales

Since 2012 until the end of June 2023 over 3.9 Mcts of combined North, Centre and South lobe diamonds have been sold for revenue of $2.2 B (average price per carat of $558/ct).

Sales lots are prepared by Lucara Botswana staff for presentation to clients, supply to Clara and HB Antwerp in a modern, ultra-secure sorting facility. Tender ales parcels and Clara designated goods conform to industry standard size ranges and descriptions.

KDM 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.

Timing of tender dates is aligned with other rough diamond sales dates to target maximum participation of buyers. Tenders of regular goods are held on a quarterly basis. 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 five 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.

Clara sales are generally held every 4-5 weeks and is dependent on supply of goods to the platform. Clients enter a bid for a polished diamond using a set of parameters that can be tailored to the client's specific demands. Diamonds are sold on an individual basis and should the client be successful they are delivered the rough diamond, with scans and polishing plans that will produce the requested polished outcome.

Prior to the middle of 2018 Lucara sold diamonds through both regular stone tenders (RST's) and exceptional stone tenders (EST's). Diamonds that qualified for EST's are rare, selected on a range of criteria including weight, quality and colour and often achieve sales prices in excess of $1 M per diamond. Lucara has discontinued selling through EST's and established an offtake agreement with HB Antwerp in 2020 with a 10-year extension signed in 2021.

19.3 Client Base

Lucara has developed a strong, geographically diverse following of clients. Lucara has 713 registered clients, demonstrating a strong interest in the KDM production for those goods sold via tender. The Clara platform has over 100 registered customers with some overlap with companies that also participate in tenders. Attendance at tenders has remained strong coming out of the global pandemic. For all +10.8 ct diamonds that are suitable for polishing Lucara has an offtake agreement with HB Antwerp.

19.4 Rough Diamond Market Outlook

The overall rough and polished markets have experienced wild fluctuations since 2019, with diamond pricing improving toward the latter part of 2019 only to experience a sharp and deep decline in pricing during the first 3 quarters of 2020 due to the global pandemic. During 2022 there was a marked and sharp increase in rough diamond prices that was greater in the quantum of value increase than the pandemic low and over a much longer period. Rough pricing declined sharply to near pre-pandemic levels and in certain categories to values observed in the pandemic trough during 2023. As a result of companies decreasing volumes of goods for sale and India imposing a voluntary ban on rough imports diamond prices began to rebound in late 2023. Pressure from lab grown diamonds continues to exert some downward pressure on smaller natural rough.

The negative rough pricing pressure combined with upward inflationary pressure across the supply chain has resulted in some mine closures and companies seeking creditor protection since 2019. These include Liqhobong and Renard, with other mines such as Gahcho Kué curtailing capital expenditures.

Current issues during 2022/23 that are applying downward pressure to the rough market include:

- Ukraine/Russia conflict, with supply if Russian goods mainly channeled through non-G7 countries:
  - Lab grown diamonds impacting smaller sizes of rough in the lower qualities, combined with negative commentary regarding natural diamonds.
- Global uncertainty caused by high inflation coming out of the pandemic;
- Political unrest in Hong Kong; and
- Lower demand from major markets such as the USA.

The longer-term outlook for natural diamond prices remains positive, anchored on improving fundamentals around supply and demand as many of the world's largest mines reach their natural end of life over the next decade. Following on the record high diamond prices achieved in early 2022, a softer diamond market emerged in the latter half of 2022 which has persisted into the second quarter of 2023, the result of global economic concerns combined with geopolitical uncertainty, including the ongoing conflict in Ukraine. Prices continued to show signs of stabilization, however, as China continues to open-up post-Covid. Sales of lab-grown diamonds increased during the period. Intense competition combined with improvements in technology continue to drive prices of lab grown diamonds down. This further differentiates this market segment from the natural diamond market and highlights the unique nature and inherent rarity of natural diamonds. The longer-term market fundamentals remain unchanged and positive, pointing to strong price growth over the next few years as demand is expected to outstrip future supply, which is now declining globally.

A strong, expanding customer base, excellent participation in tenders, growth of the Clara Platform in terms of both goods and customers, exposure to upside on final polished sales through the HB agreement on the large, high-value Type IIa in conjunction with a consistent production profile that is trending toward more higher-grade, South Lobe and EM/PK(S) production has generated a Lucara brand where the diamond price outlook is positive. Exposure to market fluctuations for the KDM production can be expected and is somewhat balanced by the multi-faceted sales mechanism deployed by Lucara.

19.5 Contracts

Excluding the diamond sales contracts discussed previously, the following are contracts that are material to Lucara that were entered into either (i) during the financial year ended December 31, 2022; or (ii) prior to January 1, 2022 that are still in effect, other than contracts entered into in the ordinary course of business:

19.5.1 Project Financing

On July 12, 2021, Lucara announced that it had signed the Facilities Agreement with a syndicate of five international financial institutions: African Export-Import Bank (Afreximbank), Africa Finance Corp., ING, Natixis, and Societe Generale in relation to a previously announced UGP Debt Financing. The Facilities, being comprised of the Project Facility and the New Working Capital Facility are being made available to Lucara Botswana by way of a senior secured term loan facility in the principal amount of up to $170,000,000 and a senior secured revolving credit facility in the principal amount of up to $50,000,000. As is typical for a facility of this type, Lucara Botswana paid for all pre-agreed fees and expenses reasonably incurred by the syndicate of five mandated lead arrangers (MLAs), as well as customary commitment and other fees in connection with making the Facilities available to Lucara Botswana. First drawdown under the Facilities occurred on September 9, 2021 following Financial Close. Lucara has drawn $125 million from the Project Loan and $15 million from the WCF. The balance in the cost overrun reserve account (the CORA) stands at $33.6 million.

The Project Facility may be used to fund the development, construction costs and construction phase operating costs of the UG expansion project as well as financing costs in relation to the Facilities.

Details of the 2021 financing information can be obtained from Lucara Diamond's 2022 annual information form. A copy of the above material contract has been filed under Lucara's profile on the SEDAR+ at www.sedarplus.ca.

Subsequent to the effective date of this Technical Report, Lucara completed a re-base of the project schedule and capital cost which are reflected in this FS update. A summary of the terms and conditions of the amendments to the Facilities Agreements reflecting the rebase are contained in press release date June 9, 2024. While the total quantum of the amount available to draw down under the Facilities has not changed, the repayment profile has been extended in line with the rebase schedule released July 17, 2023 (link). Lucara expects to continue to develop the KDM underground expansion using funds from the Project Loan (as hereinafter defined), combined with projected excess cash flow from the KDM OP mine operations and stockpiles processed during the underground construction period.

Outlined here are key terms of the Facilities Agreement, as amended:

- Up to $190 million provided to fund the development, construction costs and construction phase operating costs of the UGP as well as financing costs in relation to the Facilities;

- 8 year maturity, to June 30, 2031, with quarterly repayments commencing on September 30, 2028;

- Interest rate and Margin: LIBOR (or replacement benchmark) plus margin of 6.5% annually from Rebase Date to the Project Completion, 6.0% annually from Project Completion to June 30, 2029, and 7.0% annually thereafter;

- Commitment Fee: Lucara Botswana to pay 35% of the Margin per annum applicable to the Project Loan Facility on the Available Commitment for the Project Loan Facility;

- CORA: Amount of $61.7 million to be funded by June 30, 2025;

- First ranking security over all assets of the Borrower on a fixed and floating basis, as well as all shares in and shareholder loans into the Borrower and all shares in and shareholder loans into the intermediary companies between the Sponsor and the Borrower;

- The project facility will require interest rate hedging of at least 75% of the Borrower's exposure to be arranged as a condition subsequent to Financial Close; and

- Positive and negative covenants, including financial ratios, as well as events of default and a cash flow waterfall customary to a financing of this nature are set out in the amended Facilities agreement.

Outlined here key terms of the WCF:

- Up to $30 million for a senior, secured WCF for working capital and other corporate purposes of the Borrower;

- Interest rate and Margin: LIBOR (or replacement benchmark) plus margin of 6.5% annually for the period commencing from the date of the amendment to Projection Completion, 6.25% from Project Completion to June 30, 2029, and 7.25% annually thereafter; and

- Commitment Fee: Lucara Botswana to pay 35% of the Margin per annum applicable to the Working Capital Facility on the Available Commitment for the Working Capital Facility.

In connection with the amended Facilities, Lucara's largest shareholder, Nemesia S.a.r.l. (Nemesia), has agreed to enter into a shareholder guarantee and an amendment to the shareholder standby undertaking, in favor of the Lenders of up to $63.0 M in aggregate (collectively, the Shareholder Guarantees), which will support the UGP expansion if the projected cash flows from the KDM operations, combined with funds available from the Project Loan, are insufficient. The Shareholder Guarantees may also be drawn in the event of a shortfall in Lucara's ability to fund the CORA by June 30, 2025. As consideration for providing the Shareholder Guarantees and subject to receipt of all required regulatory approvals, Lucara will issue 1,900,000 common shares to Nemesia, subject to receipt of TSX approval and a further 7,500 common shares per $500,000 drawn, calculated monthly, should any amount be drawn under the Shareholder Guarantees, subject to TSX and other regulatory approvals (the Nemesia Consideration).

19.5.2 Project Construction Contracts

- On July 22, 2021, Lucara Botswana (Pty) Ltd. signed an Engineering, Procurement, Construction Management (EPCM) services contract for the KDM UGP with JDS Foreign Enterprises Inc., a wholly owned subsidiary of JDS Energy & Mining Inc. The contract is a time and material contract with a significant bonus and penalty components which incentivizes JDS to achieve JDS cost, Project cost and Project schedule targets; and
- On May 12, 2022, Lucara Botswana (Pty) Ltd. signed a cost-plus contract with UMS Botswana (Pty) Ltd. for the sinking and equipping of the two Project shafts as well as defined station and level lateral development immediately adjacent to the shafts. There is a significant bonus and penalty component to the contract which focuses on quarterly (approx.) schedule milestone achievement and overall contract cost.

The Project has numerous other smaller contracts for various equipment, supplies and services.

All contracts in place have terms that are within industry norms.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACTS

20.1 Environmental Studies Completed to Date

20.1.1 Historical Studies and Permitting

KDM has been in commercial operation since 2012. As noted in previous 43-101 reporting, two pre-mining environmental studies were conducted for KDM (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). As the responsible authority, the Botswana Department of Environmental Affairs approved these studies in 2008 and 2010, and subsequent updates of the EMP in 2013 and 2016. In terms of the Mining License (ML 2008/6L); 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.

20.1.2 Recent Studies and Permitting

The Botswana Ministry of Mineral Resources, Green Technology and Energy Security approved Lucara Botswana's application for a renewal of the mining license (ML2008/6L) in January 2021 for a period of 25 years. The latest update to the EMP, which was developed by Digby Wells (2020) and incorporates the UGP, was approved by the Department of Environmental Affairs (DEA) in June 2020. As part of this process, KDM also received approval for the water rights for its groundwater abstraction, dewatering of the UGP, and monitoring boreholes. A list of current licenses and permits are presented in Section 20 below.

A new EIA and its regulatory approval are still required for the proposed on-site storage and mechanical evaporation of significant volumes of produced saline groundwater (TDS ±30,000 mg/l) in a lined pond between 2026 - 2030. By 2030, additional produced water disposal plans will need to be developed for the remaining LOM. This future plan is expected to be subject to an additional EIA and its regulatory approval.

20.1.3 Capacity

The Environment, Health, Safety & Community Relations (EHS & CR) Department comprises approximately 37 positions (including four fire officers). The department includes dedicated health and safety, medical/wellness, sustainability, environmental, waste management, stakeholder engagement as well as corporate social investment line functions.

20.1.4 Environmental Management

With a producing mine, Lucara demonstrates that it follows Good International Industry Practice as evidenced below.

- Lucara is a certified Member of the Responsible Jewellery Council. Its Code of Practices Standards covers ethical, social, human rights and environmental practices, and conformance is reviewed periodically by independent auditors. Lucara's latest certification expires in March 2024 and the re-certification process has commenced;

- Lucara adopted the IFC Performance Standards and Equator Principles. As a loan recipient from Equator Principles Financial Institutions, Lucara is subject to periodic audits by Independent Environmental and Social Consultants;

- KDM maintained its ISO 45001 certification (latest surveillance audit Oct 2022) for its occupational health and safety management system, and has aligned its environmental management system with ISO 14001;

- Lucara conducted self-assessment and external verification (latest verification: Dec 2021, valid for a 3-year period ending Dec 2024) of alignment with the Mining Association of Canada's (MAC) Towards Sustainable Mining (TSM) Standards (protocols include Biodiversity Conservation, Climate Change, Crisis Management and Communications Planning, Indigenous and Community Relationships, Preventing Child and Forced Labour, Safety, Health, and Respect, Tailings Management and Water Stewardship); and

- Lucara, as best practice, is applying GISTM to FRD and CRD facilities. Lucara has engaged with an independent Engineer of Record and the first review by a three-person Independent Technical Review Board is scheduled for Q4/2023.

The 2020 EMP update sets out the mitigation measures and impact management / monitoring activities that KDM must undertake to maintain compliance during the current operational and later closure phase of the Project.

The mine continues to monitor and, for many key performance indicators, publicly disclose results in its annual sustainability reporting for the following topics (KPIs subject to external assurance marked with and asterisk (*)):

- Air quality and noise;

- Groundwater quality;

- Water use and discharge*;

- Energy use*;

- GHG emissions*;

- Waste Management*;

- Land disturbance;
- Biodiversity; and
- Environmental incidents.

Monitoring samples are typically analyzed by independent third parties. As incidents occur, they are logged, addressed and closed out in cooperation with the relevant department. Where monitoring results indicate the need for corrective actions, these are developed and implemented over time.

20.1.5 Natural Setting

The Orapa-Letlhakane region is generally flat with a slight fall towards the north / northwest. Ground elevation ranges between 1,000 m in the south / southeast and 950 m further towards the northwest. Surface drainage is virtually non-existent, except for the dry Letlhakane River (fossil valley) which drains towards the Makgadikgadi pans.

The region is characterized by a semi-arid to arid climate with hot, wet summers and cold, dry winters. The highest temperatures are experienced during summer with maximum and minimum temperature averaging above 30°C and 20°C respectively. During the winter months, the average minimum temperature often falls below 10°C. The wind direction is quite variable, especially at low speeds (<7 knots). The majority of the high-speed winds blow to the west and west-northwest.

Rainfall in the Letlhakane area is temporary and spatially variable. Typically, most rainfall occurs between September and April, although some events have been recorded between May and August. The soils of the mining lease area comprise arenosols, luvisols and calsisols, covered in mopane tree and shrub, savannah with occasional grassy areas. Most of the surface flow tends to be localized to the numerous pans dotted throughout the region. The flat landscape is altered by the presence of silcrete / ferricrete hillocks in the east, the numerous pans, especially to the west and northwest, and anthropogenic features of relatively high relief in the mining areas of Orapa and Letlhakane. These features are dumps (waste rock, tailing, slimes or slurry) rising up to approximately 60 m above the flat plain. There are two pans in the vicinity of the mine area, one to the east and one to the west.

20.1.6 Fauna and Flora

The area of the Mining License (ML 2008/6L) falls within the range of most of Botswana's savanna species. In addition to regional diamond mining activities, the area hosting KDM features farming and grazing activities. The area of the Mining License is covered by a mix of two vegetation types: mopane tree savanna on poorly drained soils with high clay content, and mopane shrub savanna on sand. The area features several species with conservation status tabulated below (Ecosurv, 2021, Lucara 2023).

Table 20-1: Species with Conservation Status at KDM and Transmission Line

Conservation Status	IUCN	Botswana	Comments
Critically Endangered	1	1	Nests of white-backed Vulture (*Gyps africanus*) were observed 1.6 km from the new transmission line
Endangered	1	1	African Elephant (*Loxodonta Africana*), common in Botswana, rarely seen within the licensed area
Threatened	0	2	Devils claw (*Harpagophytum procumbens*) and Hoodia (*Hoodia currorii*) in the license area
Vulnerable	4	4	Ground pangolin (Smutsia temminckii), Lion (Panthera leo), Cheetah (Acinonyx jubatus), Small Spotted Cat (Felis nigripes), Leopard (Panthera pardus), rarely seen within or near licensed area

Source: Ecosurv (2021), Lucara (2023)

In response, Lucara updated and is implementing a Biodiversity Management Plan and Wildlife Hazard Management Plan, added bird deflectors to the recently completed, high voltage transmission line, and reviewed opportunities to transplant the Devils claw and Hoodia should they be identified within the planned construction areas.

20.1.7 Ground Water and Water Management

Given the arid context of the KDM region, groundwater in this region is important for meeting demand (current and future) for mining, domestic supply and livestock watering. Lucara's latest annual self-reporting for the calendar years 2022 indicates annual abstraction from pit dewatering and water supply boreholes in 2022 was 1,972,582 m^3.

Section 18 of this report explains the volumes and chemical composition of the saline water (groundwater inflows) to be pumped to the surface by the future UG mine, as well as for the description of disposal options considered.

Lucara's selected option from 2026 – 2030 involves converting an existing borrow pit area located north of the UGP pad area to a lined pond where mechanical evaporators will be installed for the disposal of the saline water to be pumped to the surface. Section 18 provides a description of the planned mechanical evaporators/pond system and of its operation.

As of the effective date of this report, Lucara informed that a presentation of the management plan for the saline water was made to the Botswana Dept. of Environment. Following this presentation, the Botswana Dept. of Environment requested Lucara to have a registered Consultant perform an Environmental Impact Statement (EIS) of the plan for the disposal of saline water.

20.1.8 Climate Change and GHG

Botswana's Third (2019) National Communication to the United Nations Framework Convention on Climate Change highlights the vulnerability of the country to Climate Change. Climate scenarios were constructed for precipitation and temperature for the year 2050 based on the Representative Concentration Pathway (RCP) of 4.5 and 8.5 GCM/RCM ensemble. By 2050, most the country will experience high average temperature (25.9-26.9°C) and seasonal and annual mean precipitation showed a general decreasing trend. Botswana's first Intended Nationally Determined Contributions highlight the country's intention to achieve an overall GHG emissions reduction of 15% by 2030, taking 2010 as the base year.

KDM is connected to the national grid, which is largely supplied by the coal fueled Morupule Power Plant, supplemented by imported power from neighbouring countries, mainly South Africa. KDM also operates a diesel-fueled mobile equipment fleet. Until constructing and energizing a high-voltage transmission line supplying KDM since December 2022, up to 17 diesel generators were used to provide power for the UGP.

Lucara's 2022 Sustainability Report summarizes its third-party assured GHG emissions and intensities as tabulated below.

Table 20-2: Lucara's GHG Emissions and Intensities for 2020 to 2022

Greenhouse Gas Emissions	2020	2021	2022
Scope 1 (tCO_2e)	18,692	16,618	26,685
Scope 2 (tCO_2e)	55,443	58,494	59,116
GHG Intensity (Total CO_2e(kt)/ore + waste rock mined (t))	13.5	11.7	17.9

Source: Lucara (2023)

Lucara Botswana has developed a Decarbonization Strategy, generated a Decarbonisation Action Plan, contracted Mott MacDonald to conduct a prefeasibility study for a large-scale solar PV project for KDM, is exploring feasible options to reduce its GHG emissions by 15-30% by 2030, and commissioned a study to estimate its Scope 3 GHG emissions.

20.1.9 Tailings Management

Section 18.4 Tailings Management Facility details the (a) Coarse Residue Deposit (CRD), which comprise coarse tailings placed in designated storage areas, without containment structures such as walls or dams, and (b) FRD, which are rectangular structures contained by dams. Fine residues (also referred to as slimes) are pumped to the FRD facilities as a slurry, after which decant water can be collected and recycled.

FRD 1 has reached maximum capacity and closure plans for the facility are currently underway. Based on the results of Tailings Dam Breach Analysis (TDBA), FRD 1 was previously classified as a “High Hazard” facility in accordance with the criteria outlined in the South African National Standards 10286:1998 “Code of practice, Mine residue”, and “Very High Hazard” facility according to Global Industry Standard on Tailings Management (GISTM) criteria (Lucara, 2023).

As detailed in Section 18.4.7, TDBA analysis based on 2025 design specifications of FRD 2 recommended a “High” Consequence Classification in terms of the GISTM classification system, A proposed construction of a flood mitigation berm shown on Figure 18-8 would lower consequence classification from “High” to “Significant”.

At the time of effectiveness of this report, no TDBA had been performed for the updated design criteria (1031 masl) for FRD 2 and FRD 3, which are expected to be designed to comply with GISTM.

20.1.10 Waste Rock Storage Facility

The WRSF is located west of the FRD dam and accommodates all waste rock not used for FRD dam impoundment construction. The WRSF side slopes will be constructed to a gradient of 1:3 and the maximum vertical height of the WRSF will be 25 m.

As stipulated in the EMP, seepage run-off and dust fallout from the facility are monitored.

20.2 Socio-Economic Setting

20.2.1 Land Use

KDM is located in the Central District of Botswana, 15 km south-west of the town of Letlhakane to which it is connected via hardened surface road. Letlhakane is a regional centre in central Botswana with a number of diamond mines operating within 75 km to the west and northwest of the village.

According to the Central District Integrated Land Use Plan (CDILUP) (Geoflux, 2007), the primary use for tribal land in the sub-district is grazing. The Orapa-Letlhakane region has mixed secondary uses which include arable, settlement and mining activities. The area between Letlhakane and KDM is used for arable and grazing purposes; with grazing becoming more dominant from KDM towards the south, southwest and west. The grazing areas are mainly communal; however, commercial ranches have been demarcated further to the southwest. These ranches, though intended to improve the use and management of land resources, reduce the land available to communal farmers.

20.2.2 Social Impact Assessment

The approved EIA (Geoflux, 2007) included a Social Impact Assessment and dedicated stakeholder engagement line functions in the EHS & CR Department to manage stakeholder engagement, social aspects and obligations. Since the project commissioning, the community relations team has been engaging with local stakeholders on an ongoing basis.

As part of the KDM UG FS, the social impact of the mine and the project were separately assessed and compiled into a separate Social Impact Assessment (SIA) document which maps the following:

- The existing socio-economic impacts of the current opencast mining project;
- The likely socio-economic impacts of the proposed activities including:
  - Closure of the current opencast operation; and
  - Construction, operation and eventual closure of the proposed UG operation.
- Current and planned mitigation measures to avoid or ameliorate negative impacts and enhance positive ones.

Letlhakane has eight public schools which are four primary schools, three junior secondary and one senior secondary school. The various mines in the area provide the predominant employment activity, followed by farming. The findings of social impact studies show that economic opportunities associated with the mine’s operations and expansion. Issues of concerns voices during public consultation relate to health and safety performance, need for community development activities, presence of threatened species, access and safeguarding water resources, environmental degradation, as well as its eventual closure are the primary concern for the majority of stakeholders. Other, broader national issues of concerns relate to HIV/AIDS (Botswana has one the world’s highest infection rates), presence of Gender Based Violence (GBV), and vulnerability of Remote Area Dwellers.

Lucara Botswana’s Stakeholder Engagement Plan was revised most recently in March 2023, and provides stakeholder analysis, identifies vulnerable groups, describes approach for community development projects, and supports a structured and robust engagement program. Lucara maintains a Community Social Investment (CSI) program, investing approximately $4.3 million in 2022, and support initiatives ranging from developing a sports complex, ameliorating malnutrition through support of farming cooperatives, and GBV awareness raising.

20.2.3 In-migration (Influx)

KDM is located in a region featuring several other major diamond mining operations. These include Debswana’s Orapa, Letlhakane and Damtshaa Mines (also referred to as OLDM). A recent Debswana sponsored immigration study (Geoflux, 2022) notes that most stakeholders attribute in-migration to the existence of OLDM, with several of these operations starting several decades ago.

As part of the UGP, the second phase of a construction camp with a total capacity of up to 200 workers was completed adjacent to KDM in 2022. This is expected to reduce housing and other pressures on Letlhakane village during the construction of the UGP. Lucara Botswana's Human Resources policy, which was last revised in 2022, emphasizes the need for development and implementation of localization policies.

Lucara has been involved in a Debswana sponsored study on influx. The draft report provides recommended management measures to be implemented by Debswana. Lucara indicated its intent to support the finalized report's findings and recommendations, where possible and appropriate.

20.2.4 Sites of Archaeological and Cultural Importance

An Archaeological Impact Assessment (AIA) carried out in 2008 revealed several archaeological and burial sites within KDM and along the access road corridor. The artifacts that were discovered included stone tools, pieces of pottery, bones and glass objects. The mine committed to protecting burial sites and carried out archaeological awareness programs. The burial sites have been fenced off and periodic monitoring has been carried out during the development phases. An updated survey was undertaken in October 2018. No archaeological resources were identified during the site survey.

An additional AIA study, including a review of past reports and records, and comprehensive field survey was completed for the new TSF in 2022. The survey revealed no evidence of graves, cultural sites, archaeological sites, historical structures or buildings, within the area planned for development. Therefore, the proposed project area was graded "5" on the Botswana Department of National Museum and Monuments (DNMM) grading scale. This means that the project area has little archaeological significance and no further investigations were deemed to be necessary. The report recommended archaeological monitoring during ground disturbing activities to deal with chance finds and support awareness raising activities of contractors.

20.3 Responsible Mining and Human Rights

Lucara Diamond Corp is a signatory (participant) of the United Nations Global Compact (UNGC), which signifies commitment to promoting universal principles on human rights, labour, environment and anti-corruption. Lucara's latest 2022 Communication on Progress for the UNGC was published in June 2022.

In 2021, Lucara completed a desktop review of human rights issues that are relevant to KDM. A salient human rights issue identified during this process was the right to water.

Human rights topics are also enshrined in numerous certification and verification schemes in which Lucara participates. These include the Kimberley Process (designed to increase transparency and oversight in the diamond supply chain in order to eliminate trade in conflict diamonds), Responsible Jewellery Council (built on key development frameworks, including the Universal Declaration of Human Rights, ILO Principles, and the UNGC and Sustainable Development Goals, latest certificate valid until March 2024), and the Mining Association of Canada's Towards Sustainable Mining (MAC TSM, which promotes environmental and socially responsible mining).

Lucara's security personnel receive training on human rights principles, including the United Nations Guiding Principles on Business and Human Rights, as well as applicable local laws, and new contractors receive training on human rights, fair employment, and other relevant topics. There are no artisanal mining activities at or near KDM. Lucara maintains grievance mechanisms.

Lucara has been disclosing its annual sustainability reports since KDM's operations started in 2012. Lucara's sustainability reporting has been subject to independent assurance since 2016. Lucara has been recognized in The Globe and Mail's (Canadian newspaper) 2021, 2022 and 2023 benchmark studies of female leadership in corporate Canada, won the Junior ESG Award in the Equality and Diversity Category at the Mining Indaba 2022, won the Junior ESG Award in the Economy Category at the Mining Indaba 2023, and previously, received the 2016 PDAC Sustainability Award.

20.4 Mine Closure

Digby Wells developed the latest Mine Rehabilitation and Closure Plans and updates for KDM in line with Section 65 of the *Botswana Mines and Minerals Act* (1999) and South Africa`s National Environmental Management Act, 19998 (Act No. 107 of 1998) (Digby Wells, 2021). The mine is obliged to develop and implement a mine closure and rehabilitation plan (MCRP) 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.

In the absence of Botswana-specific closure rates, the closure liability calculation is based on annually updated master rates used for closure planning in South Africa. As is common practice on southern African mining operations at this stage of mining, the cost for water treatment is excluded. Water discharge from the mine is not expected after operations cease so no water treatment is envisioned at closure. As the mining operation and Botswana mine closure guidance evolves, the closure liability estimates will require further refinement.

Based on the local climatic and soil conditions, sustainable grazing has been identified as an appropriate post-closure land-use option. Consultation with stakeholders will be required to ensure buy-in.

Digby Wells estimated reclamation liability was approximately $30 million (Digby Wells, 2020) and increased to $34 M to bring to 2023 estimates. Lucara Botswana has provided financial guarantees totalling BWP 240.0 million for reclamation obligations, consisting of cash on deposit of BWP 40.0 million (US$3.1 million) and a BWP 200 million (US$15.4 million) standby letter of credit (First National Bank of Botswana Limited, 2022).

20.5 Permitting

Lucara maintains a register for its licenses and permits for KDM and the UGP. A list of permits held or in the process of being acquired by KDM is presented below.

Table 20-3: KDM Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIA Permit	DEA/BOD/CEN/EXT/MNE 015(7)		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 license	Dept. of Water Affairs	*Water Act*
Borehole Certificates	In Place	Valid for the duration of the mining license	Dept. of Water Affairs	*Boreholes Act*
Dumps Classification	All clarified	All dumps active	Dept. of Mines	*Mines, Quarries, Works and Machinery Act*
Surface Rights	LT/SLB/B/1 IV (231)	ML years	Ngwato Land Board	*Tribal Land Act*
Radiation License	BW0315/2021	6-Nov-25	Radiation Inspectorate	*Radiation Protection Act*
Incinerator Permit	DJM 2020/08-05	31-Aug-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Waste Water Treatment Plant	WMF01/2022/11/20-WWTW/Karowe Diamond Mine	30-Nov-24	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Landfill	WMD/22-2022/304-10/LF/Letlhakane	31-Dec-24	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Salvage yard	WMF/20-2022/20-11//Letlhakane	31-Dec-24	Department of Waste Management Pollution Control	*Waste Management Act*
Permit to purchase, acquire and Possess Explosives	F001/2022	31-Dec-24	Dept. of Mines	*Explosives Act*
Permit to carry bulk explosives	EX.10-07/2023 Vehicle No: B868BOY	31-Dec-24	Dept. of Mines	*Explosives Act*
Explosives magazine license	00003513A	31-Dec-24	DME	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	00003512A	31-Dec-24	DME	*Explosives Act*
Permit to import and possess explosives	Jan-22	31-Dec-24	DME	*Explosives Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Application for restricted blasting license		N/A	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B681BMU	23-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B693BRO	24-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B339BPM	25-Jun-2024	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B429BJB	26-Jun-2024	DOM	*Explosives Act*
License to manufacture explosives	E-PCE0410/2022 Vehicle No: B693BRO	31-Dec-2024	DME	*Explosives Act*
Box storage for conveyance and Storage of explosives	F01/22 F02/22 F03/22 F04/22	31-Dec-2024	Dept. of Mines	*Explosives Act*
Blasting License for magazine master	In Place	valid and appointment renewed annually	Dept. of Mines	*Explosives Act*
Airstrip License	B509	LICENSE NO. B509	Civil Aviation	*Aviation Act*
Generator Licenses		Once off	BERA	*BERA Act*
Solar photovoltaic plant		Once off	BERA	*BERA Act*
Standby Generator Licenses		Once off	BERA	*BERA Act*
Mining License	2008/L6	March-46	Dept. of Mines	*Mines & Minerals Act*
License to possess and use radioactive sources	BW061/2022	1-Aug-24	Radiation Protection Inspectorate	*Radiation Protection Act* (No. 22 of 2022)
Winder Engine drivers	M35 M 1 (20)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Kibble Winder 10 - 039 - Ventilation shaft	M35 M 1 (30)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Kibble Winder 10 - 069 - Production shaft	M35 M 1	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Kibble Winder 10 - 071- Ventilation shaft	M35 M 1 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Vertical Shaft Mucker (VSM)	M35 M 1 (33)	15-Oct-24	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Vertical Shaft Mucker (VSM)	M35 M 1 (14)	15-Oct-24	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Approval letter for charging units	11-May-00	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO2/22	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO3/23	N/A	Dept. of Mines	*Explosives Act*
Authorization for Explosive storage box	FO4/24	N/A	Dept. of Mines	*Explosives Act*
Mobile rescue winder - truck mounted	M35M (16)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Capacity increase for magazine No. 385	EX.5 XXII (27)	N/A	Dept. of Mines	*Explosives Act*
Drii Approval Sandvick boom drill rig	2 C 66 XXV11	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Kibble Winder 10 - 069 - Production shaft	DOM 6/13/51(8)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Kibble Winder 10 - 069 - Ventilation shaft	DOM 6/13/51(9)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Permit to purchase, acquire and Possess Explosives	E - PPAP0035/2024	31-Dec-24	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	E-PCE0161/2024	30-Jun-24	Dept. of Mines	*Explosives Act*

Source: Lucara (2024)

21 CAPITAL COST ESTIMATE

21.1 Capital Cost Summary

The capital cost estimate was prepared using a combination of first principles, applying project experience and using vendor / contractor provided budgetary quotes while avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed.

The Capital Cost Estimate consists of three parts:

- Pre-production period - Sunk Costs;
- Pre-production period - Estimated Costs; and
- Sustaining period – Estimated Costs.

30% of total capital costs are considered sunk. The remaining estimated costs is a AACE Class 3 Estimate.

Costs are expressed in US$ with no escalation unless stated otherwise. Foreign exchange rates of BWP12.50:US$1.00 and ZAR17.00:US$1.00 are used where applicable.

The estimate is based on the assumption that contractors would mobilize only once to carry out their work and are not already mobilized on site performing other work.

Total UG specific capital costs, including sunk costs and contingencies, is estimated to be $683 M. These include costs do not include current and future sustaining costs for the existing OP operations, apart from tailings expansion and site closure.

Remaining pre-production capital costs specifically associated with developing the UG amount to $387 M.

Remaining contingency for the UGP is $32 M, or 8% of remaining spend. Closure costs amount to $34 M, are included in sustaining capital, and were assumed to occur concurrent to plant closure.

These costs are summarized in Table 21-1.

Table 21-1: Capital Cost Summary

WBS	Capital Costs	Pre-Production			Sustaining (M$)	LOM Total (M$)	Weight (%)
		Sunk (M$)	Estimated (M$)	Subtotal (M$)			
1000	Mining	140.4	253.1	393.5	124.8	518.2	63%
2000	Site Development	12.7	13.4	26.1	6.6	32.7	4%
3000	Process Plant	0.0	0.1	0.1	0.0	0.1	0%
4000	Tailings and Mine Waste	0.0	0.0	0.0	42.8	42.8	5%
5000	On-site Infrastructure	13.0	5.1	18.1	0.0	18.1	2%
6000	Buildings and Facilities	2.1	3.1	5.2	0.0	5.2	1%
7000	Off-site Infrastructure	23.3	0.4	23.7	0.0	23.7	3%
8000	Project Indirects	9.4	21.7	31.1	1.4	32.5	4%
9000	Owner Costs	63.6	89.9	153.5	0.0	153.5	19%
Subtotal		**264.5**	**386.8**	**651.3**	**175.6**	**826.9**	**100%**
10000	Contingency	0.0	31.9	31.9	13.3	45.2	
11000	Closure	0.0	0.0	0.0	34.0	34.0	
Total Capital Costs		**264.5**	**418.7**	**683.3**	**222.9**	**906.1**	

Notes:

*Numbers may not add due to rounding.

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.2 Basis of Estimate

The Project pre-production capital estimate includes all costs to develop the UGP to a commercially operable status. Pre-production capital costs are inclusive of sunk costs incurred on the KDM UGP after the 2019 Feasibility Study. The sustaining capital estimate includes all costs to sustain the existing operating site (process plant and tailing facility), sustain the UGP during operations. Owner's reserve accounts are not considered in the FS estimates or economic cash flows.

The following key assumptions were made during development of the capital estimate:

- The capital estimate is based on the contracting strategy, execution strategy, and key dates described within the Project Execution Plan (PEP) described in Section 25.1 of this report;
- UG mine development activities will be performed by a contractor;

- UG mine operations activities will be performed by the Owner's Team; and
- All surface construction (including earthworks) will be performed by local contractors.

The following key parameters apply to the capital estimate:

- Estimate Class: The capital cost estimate is considered a Class 3 feasibility cost estimates;
- Total project definition is estimated to be 50% based on 30% sunk costs and remaining 70% at a 30% estimate class. (30% * 100%) + (70% * 30%) = 50%;
- Estimate Base Date: The base date of the capital estimate is Q3 2023. No escalation has been applied to the capital estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate, except when vendors have manufactured equipment using imperial units; and
- Currency: All capital costs are expressed in US$. Table 21-2 presents the exchange rates used for costs estimated in foreign currencies.

Table 21-2: Foreign Currency Exchange Rates

US$	Exchange Rates	Currency
1 US$ =	1.33	C$
	12.5	BWP
	17.0	ZAR

Source: JDS (2023) - LUCKAR14E - Cost Assumptions - RevA 2023.08

21.2.1 Quantity Development

The capital estimate has been developed largely from engineering quantities obtained from engineering drawings, lists, MTOs, and the 3D Deswik Model of the mine design. In-house benchmarks have been used where the engineering information is not sufficiently developed to prepare accurate quantities. The level of accuracy in the estimate varies depending on the engineering progress of the given scope and discipline. All quantities are "neat" growth or wastage allowances have been applied based on the degree of engineering completed and on a comparison to historical data, as outlined in Table 21-3.

Table 21-3: Standard Growth Allowances (by Discipline)

Description (Discipline)	Unit	Growth Allowance (%)
Civil and Earthworks	m/m^3	20
Concrete	m^3	10
Structural Steel	t	5
Architectural	m^2	5
Mechanical	ea	N/A
Platework	t	10
Piping	m	10
Electrical Bulks	m	10
Instrumentation Bulks	m	10
Ventilation Bulks	ea/m	10

Source: JDS (2023)

21.2.2 Equipment

Equipment costs were developed from Actual Costs, Purchase Orders, Budgetary Quotes and database costs from comparable projects. Equipment costs will consist of:

- Base Price;
- Project Specific Accessories, identified with the vendor representative;
- Capital Spares, identified as either a quoted cost or 10% of the sum of the base price and project specific accessories;
- Freight, identified as either a quoted cost or 5% of the sum of the base price and project specific accessories;
- Assembly / Vendor Support, identified as either a quoted cost or 1.5% of the sum of the base price and project specific accessories; and
- First Fills, identified as either a quoted cost or 0.5% of the sum of the base price and project specific accessories.

21.2.3 Materials and Consumables

Materials and consumable costs were developed from a combination of Actual Cost, Purchase Orders, Budgetary Quotes, and database costs from comparable projects. Materials and consumable costs are used to derive first principal cost estimations where contractor pricing has not been provided.

Material unit costs have been sourced from local suppliers where possible and otherwise estimated by JDS based on experience.

21.2.4 Installation

Direct field installation manhours for permanent infrastructure installations have been estimated by JDS based on the following methods:

- Proposals and budgetary estimates from Southern African contractors;
- Constructional schedules from comparable installations; and
- Direct installation hours per material quantity estimated from JDS project experience.

The direct field installation manhours reflect the Southern African weather conditions. Labour provisions consider a larger component of helpers and personnel in training than is otherwise considered for North American operations. Construction equipment hours, costs and fuel consumption, is estimated based on factors applied to the direct field installation manhours.

21.2.5 Labour

UG mining staffing levels are built up based on the productivities (man-hours) required for capital development and installation activities occurring within a given time period. As such, mining manpower fluctuates throughout the capital development period.

Lucara uses a tiered salary system, where fully burdened rates include provision for the following:

- Base Salary;
- Shift and Standby Allowance;
- Housing, Gratuity, Medical and Insurance;
- Leave Allowance;
- Cell Phone and Car Allowance; and
- UG worker premium.

The labour workforce responsible for construction will be almost entirely contracted, less existing on-site Owner’s team management. Expatriate labour rates are benchmarked against the current shaft sinking contractor rates. The mine plan envisions primary contractors working to develop the mine with support of additional sub-contractors to manage specific procurement packages.

In general contractor mark-ups assumed are outlined in Table 21-4.

Table 21-4: Contractor Services Mark-Up

Description	Unit	Value	Comments/Source
Contractor Markup on local labour	%	15.0	Mark-up of Lucara labour rates[1]
Contractor Markup for Expat positions	%	40.0	In addition to local contractor mark-up
Contractor Markup on equipment OPEX	%	15.0	Main parts, tires, chains, etc.
Contractor Markup for Equipment Ownership	%	15	% of capital charged monthly with cap of 450 hours/month
Contractor Markup on Drill consumables	%	15	Drilling consumables only

Note:

[1] See Table 22-2 for local labour budgets.

Source: JDS (2023)

Annual contractor salaries are based on working two 12-hour shifts per day and account for all overtime, travel, and burdens. Burdens amount vary by salary tier and account for approximately 50% of the base salary.

A summary of the labour requirements is located in Figure 16-70. Shaft and raise bore labour requirements have been provided by contractor estimates. Development, drill and blast, Alimak, and UG construction services contractor labour requirements have been estimated from first principals.

21.3 Mining

Mining capital costs consist of primarily UG Mining activities, there are no capital costs associated with OP capital projects. Sunk costs include surface infrastructure to support UG mining, as well as the shaft sinking infrastructure and the portion of both shafts that have been constructed. Primarily the remaining UG development and infrastructure costs, as well as the sustaining costs associated with UG Mining have been estimated.

UG capital costs contain a mix of actual costs, purchase orders, budgetary quotes from local vendors where possible, and database costs from comparable projects. Time related costs for development or infrastructure installations have been estimated by JDS or by third party vendors and contractors.

UG capital costs are summarized in Table 21-5.

Table 21-5: UG Mining Capital Costs

WBS	UG Mining Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1100	Surface Infrastructure	35.5	26.3	0.0	61.8	12
1200	Shaft Sinking and Infrastructure	100.6	90.6	0.0	191.2	37
1300	UG Development	0.0	66.5	35.3	101.8	20
1400	UG Equipment	2.8	26.2	36.8	65.8	13
1500	UG Infrastructure	1.4	36.2	10.5	48.1	9
1600	Capitalized UG Operating Costs	0.0	7.4	0.0	7.4	1
1700	Infrastructure Sustaining	0.0	0.0	42.2	42.2	8
1000	**Total Mining**	**140.4**	**253.1**	**124.8**	**518.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.3.1 Surface Infrastructure

Surface infrastructure capital costs are summarized in Table 21-6.

Table 21-6: Mine Capital - Surface Infrastructure

WBS	Surface Infrastructure	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1110	Hoist House	16.3	17.9	-	34.1	6
1120	Production Shaft Head Frame	6.0	0.1	-	6.1	1
1130	Services	0.5	-	-	0.5	-
1140	Ventilation Shaft Head Frame	2.7	-	-	2.7	1
1150	Electrical Supply and Distribution	5.7	0.8	-	6.5	1
1160	Ventilation and Cooling	2.2	6.6	-	8.7	2
1170	Compressed Air	0.8	0.3	-	1.2	-
1180	Buildings and Facilities	0.3	0.2	-	0.6	-
1190	Construction Support Facilities	1.0	0.4	-	1.4	-
1100	**Total**	**35.5**	**26.3**	**-**	**61.8**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

Surface infrastructure to support shaft sinking operations has been completed, this includes sinking hoists, headframes, electrical distribution at the UG pad, ventilation and cooling required for sinking, compressed air infrastructure and other supporting facilities.

Costs associated with the construction of the man and materials winder and the Bulk Air Cooler, which are required to support the final stages of construction and the start of operations are the major remaining surface infrastructure capital costs. The majority of these costs have been defined through purchase orders or budgetary quotes specific to the remaining tasks.

21.3.2 Shaft Sinking and Infrastructure

Shaft Sinking and Infrastructure capital costs are summarized in Table 21-7.

Table 21-7: Mine Capital – Shaft Sinking and Infrastructure

WBS	Shaft Sinking	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1210	Common Preliminaries and Generals	21.6	-	-	21.6	11
1220	Shaft Pre-Sink	5.2	0.2	-	5.4	3
1240	Shaft Main Sink	52.8	74.6	-	127.4	67
1260	Shaft Equip and Commission	0.3	10.1	-	10.4	5
1280	Shaft Operating Indirects	20.6	5.7	-	26.3	14
1200	**Total**	**100.6**	**90.6**	**-**	**191.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

The pre-sink of both the P/S and V/S are complete, with the main sink activities being currently performed by the shaft sinking contractor. Cost to complete is derived from an existing contract price prepared by the sinking contractor, updated to account for approved change orders.

Shaft Operating Indirect costs are largely shaft sinking power. In early 2023, the UGP transitioned off diesel generators onto grid power, future estimated power costs are much lower than those sunk to date on the project.

21.3.3 UG Development

UG development capital costs are summarized in Table 21-8.

Table 21-8: Mine Capital - UG Development

WBS	UG Development	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1370	Vertical Development	-	5.9	3.1	9.0	13
1380	Development Contractor General	-	60.6	35.3	60.6	87
1300	**Total**	**-**	**66.5**	**38.3**	**69.5**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

UG development includes all work completed by the development, raise bore, long hole drill and blast and Alimak contractors. UG development does not include shaft sinking which is covered in Section 21.3.2.

Costs for the raise bore and Alimak contractor mobilizations were provided by budgetary quotes. Cost for development and long hole drill and blast contractor mobilizations were estimated based on the following criteria:

- $125,000 allowance for temporary facilities;
- $2,000 per contractor to account for transport, induction training, and PPE;
- 5% Freight costs of mobile equipment; and
- $240,000 allowance for first fills.

Development costs account for the labour, equipment, materials, fuel, and supervision required to drive all lateral and vertical development prior to commercial production.

Lateral and vertical development unit costs are summarized below:

- Lateral development blended unit cost - $4,864/m:
  - Heading sizes range from 5 m W x 5 m H to 7.5 m W x 5.5 m H.
- Raise bore development:
  - Pilot 381 mm - $194/m;
  - Reaming 2.1 m diameter - $565/m;
  - Reaming 3.1 m diameter - $870/m; and
  - Reaming 4.1 m diameter - $1,167/m.

- Long hole raise (3.0 x 3.0 m) – 3,598/m; and
- Alimak raise (3.0 x 3.0 m) - 7,874/m.

21.3.4 UG Equipment

UG mobile equipment capital costs are summarized in Table 21-9 and exclude shaft equipment.

Table 21-9: Mine Capital – UG Equipment

WBS	UG Equipment	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1410	Drilling	1.6	22.8	14.7	39.1	59
1420	Charging	-	-	0.2	0.2	-
1430	Loading	-	1.4	13.3	14.8	22
1440	Hauling	-	-	-	-	-
1450	Ground Support	-	-	1.8	1.8	3
1460	Services	0.9	0.5	0.5	2.0	3
1470	Ancillary	0.2	0.1	2.5	2.9	4
1480	Technical Services Equipment	-	1.3	3.8	5.1	8
1400	**Total**	**2.8**	**26.2**	**36.8**	**65.8**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

UG mining equipment quantities and costs were determined through a build-up of mine plan quantities and associated equipment utilization requirements. Quotes were received from local vendors and applied to the required quantities.

Mobile equipment required for production drilling, loading, and hauling will be purchased by the Owner given the unique specifications required and life of mine needs. This equipment will be purchased with 20% down payment 12 months in advance of requirement, with the remaining 80% paid upon delivery. Equipment will be brought on site three months in advance of being required UG. Mobile equipment purchases are assumed new.

Mobile equipment required for initial mine development contracts will be supplied by a development contractor(s) with the cost of ownership charged to the Owner as a monthly rate. These costs are carried within the development contract WBS 1370 and 1380.

At termination of the development contract a small portion of the fleet will be purchased from the contractor at an assumed 50% cost of new for owner operations to perform ongoing tunnel maintenance and rehab. This cost is covered here as sustaining capital.

All mobile equipment will come supplied with the necessary safety features required by local and mine site regulations including roll-over protection, fire protection, and emergency steering. Where possible, equipment will be outfitted with enclosed cabins and air conditioning to protect against heat stress. Auto-lubrication and foam filled tires will be applied where possible to reduce wear on equipment. The production LHDs will be equipped with tele remote capabilities to allow operators to sit in a control room away from the drawpoint hazards.

A mid-life major overhaul is budgeted for all equipment equal to 60% of the base price of the unit. Equipment will be replaced with new units at the end of the expected equipment life. Equipment will not be replaced within one year of mine closure and will instead be operated at a higher cost of maintenance.

Table 21-10 lists the LOM equipment purchases, rebuilds, and replacements, as well as fleet purchased to date.

Table 21-10: Mine Equipment Capital Costs

Equipment	Total Required	On Site	Contractor Supply	Owner Supply	Unit Cost (M$)	LOM Purchases	LOM Rebuilds	LOM Replacements
Surface FEL (15t/5.4 m^3)	1	1	1	-	-	-	-	-
Surface Truck (39 t)	4	-	4	-	-	-	-	-
Surface Loader Crane	1	1	-	-	-	-	-	-
Surface Tractor	1	1	-	-	-	-	-	-
Surface Telehandler 24 T	2	2	-	-	-	-	-	-
Surface Telehandler 10 T	2	2	-	-	-	-	-	-
Surface Warehouse Forklift	1	1	-	-	-	-	-	-
LHD (7t/2.8 m^3)	1	1	1	-	-	-	-	-
LHD (17t/7.0 m^3)	4	-	4	-	-	-	-	-
LHD (21t/8 m^3)	3	-	-	3	2.2	3	6	3
Jumbo - 2 Boom	4	1	3	-	-	-	-	-
Longhole Drill - ITH (Dev)	1	-	1	-	-	-	-	-
Longhole Drill - ITH (Prod)	5	-	-	5	1.4	5	5	-
Bolter	3	-	3	-	-	-	-	-
Cable Bolter	2	-	2	-	-	-	-	-
Shotcrete Sprayer	4	4	-	-	-	-	-	-
Small Explosives Truck	2	-	2	-	-	-	-	-
Large Explosives Truck	2	-	-	2	0.4	2	-	-
Transmixer	1	-	1	-	-	-	-	-
Scissor Lift	3	-	3	-	-	-	-	-
Fuel/Lube Truck	1	-	1	-	-	-	-	-
Mechanics Truck	2	-	2	-	-	-	-	-
Electrician Truck	1	-	1	-	-	-	-	-

Equipment	Total Required	On Site	Contractor Supply	Owner Supply	Unit Cost (M$)	LOM Purchases	LOM Rebuilds	LOM Replacements
Boom Truck	1	-	1	-	-	-	-	-
Grader	1	-	-	1	0.4	1	-	-
Mobile Rock Breaker	1	-	-	1	0.8	1	1	1
Stationary Rock Breaker - 41 kW	2	2	-	-	-	2	-	-
Telehandler UG	3	2	1	-	-	1	3	-

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG - r2

21.3.5 UG Infrastructure

UG infrastructure capital costs are summarized in Table 21-11.

Table 21-11: Mine Capital - UG Infrastructure

WBS	UG Infrastructure	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1510	Electrical Distribution	0.6	5.6	0.9	7.1	15
1520	Material Handling	0.4	11.1	0.5	12.0	25
1530	Maintenance Shop and Services	-	1.7	0.1	1.8	4
1540	Miscellaneous Infrastructure	-	-	-	-	-
1550	Mine Dewatering	0.4	4.0	6.3	10.8	22
1560	Mine Ventilation	-	7.3	0.8	8.1	17
1570	Piping	-	4.9	0.1	5.0	10
1580	Instrumentation and Communication	-	0.8	-	0.8	2
1590	Safety	-	0.7	1.8	2.4	5
1500	**Total**	**1.4**	**36.2**	**10.5**	**48.1**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

All major infrastructure costs were developed from a mix of actual costs, purchase orders, budgetary quotes from local vendors where possible, and database costs Sunk costs are primarily associated with purchase orders placed on long-lead time equipment. Allowances have been made for miscellaneous items. Acquisition of UG infrastructure is timed to support the mine plan requirements.

Costs captured within the electrical distribution category include all UG electrical distribution extending from the shafts. The scope primarily consists of medium voltage switchgear, Mine Power Centers, low voltage switchgear, distributions panels, small lighting transformers, equipment starters, lighting, and laterally run power cables. Network cabinets, PLCs and other instrumentation installed within the UG substations have also been included in the electrical distribution category, while the reticulation of the communications system is excluded and captured separately.

The materials handling costs include the UG crusher and conveyor as well as all mechanical equipment, structural steel, concrete, and mechanical components of the system. Costs associated with electrical installations and chamber excavation are carried elsewhere.

The maintenance shop and services include a multi-bay workspace to perform maintenance and repair, refueling, lubrication and washing, as well as store parts and consumables. The cost of excavation and ground support has been captured under the lateral development capital costs. Maintenance facility capital costs include the supply and install of all floor preparations, overhead cranes, fuel stations, service stations, fire suppression, and tooling. Auxiliary Infrastructure such as lunchrooms, ablutions, and magazines are captured within the Miscellaneous Infrastructure category.

The UG dewatering facilities include the supply and install of all furnishings including concrete, piping, catwalks, chain hoists, and beam trollies and pumps for UG sumps and booster stations.

Mine ventilation includes the supply and install of fans, ducting, instrumentation and controls, man doors, fire doors, air locks, fan bulkheads, and regulators. Time and material for blocking around the doors are included. Costs associated with electrical installations and chamber excavation are carried elsewhere.

Piping estimates consist of all piping, valves, coupling, flanges, spools for compressed air, service water, potable water, fire water and dewatering services. Piping quantities were determined from the mine plan.

The UG communications systems costs account for leaky feeder, fiber and private LTE system to support tele remote operations at the 310 L.

The safety system estimates include five (5) portable 20-person refuge chambers and one permanent 50-person refuge. Safety equipment has been priced locally and includes mine rescue equipment training and safety monitoring equipment.

21.3.6 Capitalized UG Operating Costs

Capitalized operating costs refer to expenses incurred before the start of UG commercial production and includes all activities directly related to the drilling, blasting, loading, and hauling of ore to the processing facility and waste to the storage facility.

Table 21-12: Mine Capital - UG Infrastructure

WBS	Capitalized Operating Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1610	Production Stoping	-	7.4	-	7.4	100
1600	**Total**	**-**	**7.4**	**-**	**7.4**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.3.7 UG Infrastructure Sustaining Costs

UG Infrastructure sustaining costs refer to ongoing repair, maintenance, and replacement costs for infrastructure not catered for in operating costs. Costs would include items such as period replacement of corroded pipes, liner replacement in chutes, pump rebuilds, and hoist rope replacements.

Table 21-13: Mine Capital - UG Infrastructure Sustaining

WBS	UG Infrastructure Sustaining	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
1710	Surface Facilities	-	-	8.1	8.1	19
1720	Shafts	-	-	25.0	25.0	59
1730	Drifts Rehabilitation	-	-	-	-	-
1740	Piping	-	-	3.2	3.2	8
1750	Crushing and Conveying	-	-	1.5	1.5	4
1760	Pumping Systems	-	-	2.9	2.9	7
1770	Ventilation Systems	-	-	1.0	1.0	2
1780	EC&I	-	-	0.5	0.5	1
1790	Other Sustaining	-	-	-	-	-
1700	**Total**	**-**	**-**	**42.2**	**42.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

Infrastructure sustaining costs are estimated based on a percentage of the pre-production capital cost of the infrastructure, applied annually, according to Table 21-14 below.

Table 21-14: UG Sustaining Capital Details

WBS	UG Infrastructure Sustaining	Factor	Annual Expenditure
		(%)	(M$)
1710	Surface Facilities	1	0.62
1720	Shafts	1	1.91
1730	Drifts Rehabilitation*	1	-
1740	Piping	5	0.25
1750	Crushing and Conveying	1	0.12
1760	Pumping Systems	5	0.22

WBS	UG Infrastructure Sustaining	Factor	Annual Expenditure
		(%)	(M$)
1770	Ventilation Systems	1	0.07
1780	EC&I	5	0.04
1790	Other Sustaining	1	0.62

Note:
*Rehabilitation is covered under sustaining development costs (WBS 1380).

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.4 Site Development

Site development costs include the construction of a surface dewatering pond and mechanical evaporation plant, grouting of historic exploration holes which intersect UG mine workings, Kimberlite depressurization drill programs, and remaining bulk earthworks required to equip the shafts on surface.

Table 21-15: Site Development Costs

WBS	Site Development	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
2100	Bulk Earthworks	10.4	0.6	-	11.0	34
2200	Site Roads	-	-	-	-	-
2300	Surface Water Management	0.1	6.2	6.6	13.0	40
2400	Dewatering	-	4.5	-	4.5	14
2500	Core Hole Drilling	2.1	2.1	-	4.3	13
2000	**Total Site Development**	**12.7**	**13.4**	**6.6**	**32.7**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.5 Processing Capital Cost Estimate

The processing of ore from UG is not anticipated to have a material change on the overall plant design or operation. A cost for additional metal detection has been included in the pre-production estimate.

Table 21-16: Process Costs

WBS	Process	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
3100	Plant Upgrade	-	0.1	-	0.1	100
3000	**Total Process Plant**	**-**	**0.1**	**-**	**0.1**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.6 Tailings

The FRDs will be expanded to accommodate the additional ore to be processed as part of the UGP. Construction of the first expansion is underway in 2023 and will continue through the life of mine as storage requirements dictate. Design details are discussed in Section 18.4.

Budgetary capital estimates for the FRD expansion have been generated by KDM.

KDM does not plan for any capital projects at the Coarse Residue Facility nor the Waste Rock Storage Facility.

Table 21-17: Residue Storage Facility Costs

WBS	Residue Storage Facility	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
4100	FRD - Slimes	-	-	42.8	42.8	100
4200	FRD - Coarse	-	-	-	-	-
4300	Waste Rock Storage Facility	-	-	-	-	-
5000	**Total Tailings**	**-**	**-**	**42.8**	**42.8**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.7 On-site Infrastructure

On-site Infrastructure capital costs include the supply and commissioning of the emergency backup power generator facility, surface power distribution infrastructure to the bulk air cooler, evaporation pond, and permanent winders, power factor correction equipment, surface water distribution lines, and Control Room building and infrastructure.

Table 21-18: On-site Infrastructure Costs

WBS	On-Site Infrastructure	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
5100	Electrical Supply and Distribution	10.2	2.9	-	13.1	73
5200	Water Supply, Distribution, and Treatment	2.3	0.1	-	2.4	13
5300	Waste Collection and Treatment	0.5	-	-	0.5	3
5400	IT and Communications	-	2.1	-	2.1	12
5000	**Total On-site Infrastructure**	**13.0**	**5.1**	**-**	**18.1**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.8 Buildings and Facilities

Buildings and facility costs include remaining offices, ancillary buildings, change houses, and mine rescue center construction and upgrades.

Table 21-19: Buildings and Facilities Costs

WBS	Building and Facilities	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
6100	Training Center	-	-	-	-	-
6200	Workshop and Warehouse	0.2	0.1	-	0.2	4
6300	Mine Rescue Centre	1.4	1.3	-	2.7	51
6400	Offices	0.6	1.7	-	2.3	44
6500	Change house	-	-	-	-	-
6600	Access, Fencing, and Traffic Management	-	-	-	-	-
6000	**Total Buildings and Facilities**	**2.1**	**3.1**	**-**	**5.2**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.9 Off-Site Development

Off site development costs are largely complete with remaining budget allocated to close out and maintain the power transmission line and off-site accommodation facilities.

Table 21-20: Off-site Development Costs

WBS	Off-Site Development	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
7100	Power Transmission Line	18.9	0.2	-	19.1	80
7200	Off-site Accommodations	4.5	0.2	-	4.7	20
7000	**Total Off-site Development**	**23.3**	**0.4**	**-**	**23.7**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.10 Project Indirects

Project indirects cover camp catering, office rentals, bussing, and charter flights for personnel. Also included are freight and freight forwarding services, civil material testing, and waste rock haulage from the project area to the waste rock dump.

Table 21-21: On-site Infrastructure Costs

WBS	On-Site Infrastructure	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
8100	On-site Contract Services	4.0	17.4	-	21.4	66
8200	Temporary Facilities and Utilities	-	0.1	-	0.1	-
8300	Contractor Indirects	0.2	0.5	1.4	2.1	6
8400	Freight	4.7	2.4	-	7.1	22
8500	Temporary Accommodations and Expenses	0.5	1.4	-	1.9	6
8000	**Total Project Indirects**	**9.4**	**21.7**	**1.4**	**32.5**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.11 Owner's Cost

Owner's costs are classified as the management, oversight and site operation costs that are instrumental to develop the UGP. These costs are capitalized during the construction phase. Any Owner's costs that continue beyond the project phase are then incorporated into the site G&A operating costs.

Owner's costs include:

- Engineering, Procurement, and Construction Management (EPCM) services;
- Owners labour;
- 3rd party engineering services;
- Free issue materials including fuel, power, explosives, and cement;
- Project taxes and insurance;
- Human Resources;
- Pre-production operational charges; and
- Equipment fleet maintenance.

Table 21-22: Owner's Costs

WBS	Owners Capital Costs	Pre-Production		Sustaining (M$)	LOM Total (M$)	Weighting (%)
		Sunk (M$)	Estimated (M$)			
9100	Pre-Production General and Administration	5.7	17.3	-	23.0	15
9200	Operational Charges	11.8	8.0	-	19.8	13
9300	Engineering, Procurement, and Construction Management	35.9	38.3	-	74.2	48
9400	Equipment Supply and Maintain	0.7	0.6	-	1.3	1
9500	Free Issue Materials	9.6	25.7	-	35.3	23
9600	Stay-In-Business Annual Budgets	-	-	-	-	-
9000	**Total Owners Costs**	**63.6**	**89.9**	**-**	**153.6**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

21.12 Closure Cost Estimate

Lucara Botswana has provided financial guarantees totalling BWP 240.0 million for reclamation obligations, consisting of cash on deposit of BWP 40.0 million and a BWP 200 million standby letter of credit. Closure costs were originally prepared by Digby Wells in 2019 in preparation of the 2019 Feasibility Study and encompass the entire KDM site inclusive of the UGP. UGP closure costs have been estimated using a unit rate approach against the planned UGP infrastructure. Demolition and civil contractor quotes were used where possible for the original 2019 estimate and updated to 2023 rates by using a five-year historic escalation rate of 5.3% (World Data, 2023).

Table 21-23: Closure Costs

WBS	Closure Costs	Pre-Production		Sustaining	LOM Total	Weighting
		Sunk	Estimated			
		(M$)	(M$)	(M$)	(M$)	(%)
11110	Pit Buildings and Surface	-	-	4.2	4.2	12
11120	Open Pit and Dumps	-	-	13.0	13.0	38
11130	Slimes and Dams	-	-	8.7	8.7	25
11140	UG	-	-	2.3	2.3	7
11150	Monitoring	-	-	3.0	3.0	9
11160	Project Management	-	-	2.8	2.8	8
11000	**Total Closure Costs**	**-**	**-**	**34.0**	**34.0**	**100**

Source: JDS (2023) – LUCKAR14E_FS_CAPEX SUM_r3

22 OPERATING COST ESTIMATE

22.1 Operating Cost Summary

As the KDM is currently in operations, the operating cost estimates for processing, OP mining, G&A, and cost of sales were prepared using forecast operating budgets provided by Lucara. The UG mining operating costs were prepared using first principles, and budgetary quotations. Inputs are derived from engineers, contractors and suppliers who have provided similar services to other projects.

Operating costs in this section of the report include mining, processing, coarse and fine residue deposition, and administration up to the production of diamonds from site. Off-site, in-country corporate costs such as Lucara Botswana management, cost of sales, and costs associated with Clara have been provided by Lucara and are included as sales and corporate costs. UG mine operating costs incurred during the construction phase are capitalized and form part of the capital cost estimate. All other operating costs incurred during the construction phase to support the current operations are included as part of operating costs.

Operating costs are presented in 2023 US dollars on a calendar year basis. No escalation or inflation is included. Total operating costs over the life of mine are $1,721 M and are summarized in Table 22-1.

Table 22-1: Breakdown of Estimated Operating Costs

Operating Costs	Average Annual(1)	Life of Mine	Tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Open Pit Mining Costs	24.2	72.6	5.5	13.2	4
UG Mining Costs	29.5	413.2	37.0	11.2	24
Rehandle Costs	3.4	23.6	9.7	2.4	1
Process Costs	24.7	493.7	52.2	9.5	29
Other Power Costs	5.3	105.2	52.2	2.0	6
G&A	18.3	365.8	52.2	7.0	21
Cost of Sales	4.4	87.9	52.2	1.7	5
Corporate Charges (Botswana)	8.0	159.2	52.2	3.1	9
Total	**86.1**	**1,721.1**	**52.2**	**33.0**	**100**

Notes:
(1) Average cost per year in which costs occur.
(2) Tonnes processed in relation to operating cost.
Source: Lucara (2023) - Karowe FS Model V1.7

Operational labour rates have been estimated by applying legal and discretionary burdens against KDM base labour rates. Wage scales were defined and applied to the various operational positions based on skill level and expected salary for existing site roles, consistent with current operational practice. Lucara Botswana human resources personnel were involved in the buildup and verification of the operational labour rates. Labour rates were benchmarked against similar positions in regional mines as well as existing on-site contractor rates.

Main operating costs component assumptions are shown in Table 22-2.

Table 22-2: Main Cost Assumptions

Item	Units	Base	Source
Exchange Rates, Escalation, and Taxes			
South African Rand	xZAR:1USD	17.00	KDM
Botswana Pula	xBWP:1USD	12.50	KDM
Escalation Rate	%	0%	KDM
Value Added Tax (VAT)	%	14%	BURS
Power			
Fixed Charge	BWP/month	92.78	BPC Line Power Delivered to site, excluding VAT
Demand Rate	BWP/kW	208.29	
Energy Charge	BWP/kWh	0.71	
Fuel			
Diesel Fuel, 50 ppm	BWP/L	15.06	Actuals 2023. Petrohyper Delivered to site, excluding VAT
Labour			
A2	BWP/month	130,787	KDM 2023 budgets, mid-range, fully burdened
B1	BWP/month	168,251	
B2	BWP/month	186,750	
B3	BWP/month	228,117	
B4	BWP/month	274,265	
C1	BWP/month	397,881	
C2	BWP/month	498,017	
C3	BWP/month	642,688	
C4	BWP/month	787,820	
D1	BWP/month	894,021	
D2	BWP/month	1,090,575	
D3	BWP/month	1,343,174	
D4	BWP/month	1,523,622	
E	BWP/month	1,817,693	

Source: JDS (2023) - LUCKAR14E - Cost Assumptions - RevA 2023.08

22.2 Mine Operating Cost Estimate

22.2.1 OP Operating Costs

KDM currently operates an OP mine. OP mine operating costs are based on past performance, current budgets, and account for any forecasted adjustments to the OP operating strategies.

OP operating costs are based on the five-year budget prepared by Lucara in September 2023. OP operations are currently performed by a mining contractor. The existing contract mining rates and the five-year budget were used to forecast OP operating costs beyond 2024. Incremental costs for mining at depth and haulage to WRSF destinations were applied according to the existing contract and the combined OP and UG mining schedule was used to determine the rehandling requirements and costs. The OP mining operating costs are listed in Table 22-3.

Table 22-3: Open Pit Mining Operating Cost Summary by Activity

Operating Costs	Average Annual(1)	Life of Mine	tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Open Pit Mining	24.2	72.6	5.5	13.2	75
Stockpile Rehandle	3.4	23.6	9.7	2.4	25
Total	**13.7**	**96.2**	**52.2**	**1.8**	**100**

Source: Lucara (2023) - Karowe FS Model V1.7

22.2.2 UG Operating Costs

UG operating costs refer to expenses incurred after the start of UG commercial production and includes all activities directly related to the drilling, blasting, loading, and hauling of ore to the processing facility and waste to the storage facility.

The UG mining operating costs include the following functional areas:

- Production – costs related to the ITH drilling, blasting, and mucking of ore;
- Crushing and Hoisting – costs related to the operation and maintenance of the UG crusher, conveyor, and shaft equipment, as well as surface haulage equipment;
- Mine Maintenance – costs related to the maintenance of UG fixed and mobile equipment;
- Mine General – costs related to mine support activities such as supervision, technical services, shared infrastructure, support equipment, and material delivery UG; and

- Contingency – a 10% nominal cost applied to all areas of mine operating costs.

Costs associated with UG vertical and lateral development are captured in initial and sustaining capital costs and are not included in operating costs.

Table 22-4: UG Mining Operating Cost Summary by Activity

Operating Costs	Average Annual(1)	Life of Mine	tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Drill and Blast	5.0	65.2	37.0	1.76	16
Drawpoint Operations	3.9	51.6	37.0	1.40	13
Crush and Convey (UG)	0.8	10.5	37.0	0.28	3
Shaft Operations	4.2	55.4	37.0	1.50	13
Surface Haulage	5.3	69.9	37.0	1.89	17
Mine Maintenance	2.4	32.0	37.0	0.87	8
Mine General	6.9	89.7	37.0	2.43	22
Contingency	2.9	37.4	37.0	1.01	9
Total	**31.4**	**411.8**	**37.0**	**11.14**	**100%**

Notes:

(1) Average costs apply to the year in which costs occur. Total average will not equal the sum of the average costs due to timing of expenses; and

(2) Tonnes processed are equal to tonnes mined less mine recovery.

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG - r2

Table 22-5: Mining Operating Cost Summary by Area (excluding mine G&A)

Operating Costs	Average Annual(1)	Life of Mine	tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Labour	10.4	145.6	37.0	3.94	35
Equipment	6.5	90.4	37.0	2.45	22
Materials	0.9	12.1	37.0	0.33	3
Explosives	2.9	14.4	37.0	0.39	3
Fuel	3.1	44.1	37.0	1.19	11
Power	4.8	67.8	37.0	1.83	16
Contingency	2.7	37.4	37.0	1.01	9

Operating Costs	Average Annual[1]	Life of Mine	tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Total	**29.4**	**411.8**	**37.0**	**11.14**	**100**

Notes:

[1] Average costs apply to the year in which costs occur. Total average will not equal the sum of the average costs due to timing of expenses; and

[2] Tonnes processed are equal to tonnes mined less mine recovery.

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG - r2

22.2.2.1 UG Mining Labour

UG mining staffing levels are built up based on the productivities (man-hours) required for the scheduled mining activities. As such, mining manpower fluctuates throughout the mine life.

UG labour rates are based on the existing OP labour force plus a 25% mark-up for an UG allowance. Rates include all overtime and burdens associated with 12-hour shifts. Burdens amount to 47% of the base salary, and account for items such as housing, gratuity, medical, vacation, group insurance, and for those eligible, cell phone and car allowance. Expatriate labour rates have been benchmarked against publicly available UG miner salaries within South Africa to ensure that KDM will be able to attract the talent required for specialty positions.

Table 22-6: UG Labour Cost Summary

Labour Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Drill and Blast	2.4	31.2	0.84	19
Drawpoint Operations	0.9	11.7	0.32	7
UG Crush and Convey	0.3	3.3	0.09	2
Shaft Operations	1.9	24.4	0.66	15
Surface Haulage	-	0.5	0.01	-
Mine Maintenance	2.1	27.5	0.74	17
Mine General	3.6	46.9	1.27	29
Contingency	1.1	14.6	0.39	9
Total Mining OPEX	**12.2**	**160.1**	**4.33**	**100**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

UG labour costs consider that mine management and technical roles that are currently employed by KDM will be transferred to the UG mine prior to commercial production. These become part of the Mine General OPEX.

A summary of labour positions by category during Stage 1 and Stage 2 of the mine plan is provided in Table 22-7.

Table 22-7: UG Mine Operating Labour Requirements

Operating Cost Labour	Units	Labour Type	Roster	Peak	Steady Stage
Staff					
Mine General	#	Local	5x2, 4x4	20	20
Technical Services	#	Local	5x2	30	30
Drawpoint Operations	#	Local	4x4	40	40
UG Crush and Convey	#	Local	4x4	16	16
Shaft Operations	#	Local / Expat	4x4	56	56
Surface Haulage		Local	4x4	26	26
Maintenance		Local	4x4	51	53
Contractors					
Development Contractor*	#	Expat / Local	5x2, 2x1	187	0
Drill and Blast Contractor	#	Expat / Local	5x2, 2x1	118	118
Raisebore Contractor*	#	Expat	2x1	31	0
Alimak Raise Contractor*	#	Expat	5x2, 2x1	17	0
Total Employed					
Staff	#			239	241
Contractors	#			353	118
Total	**#**			**592**	**359**
Day Shift	#			207	130
Night Shift	#			121	71
On-Site (Day + Night Shift)	**#**			**327**	**200**

Note:
*Costs are capitalized. Shown only to provide indication of peak underground workforce.

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

22.2.2.2 UG Mining Equipment

UG mining equipment usage costs are based on the equipment operating hours required to meet the life of mine plan. Operating hours are derived through first principal build-up using productivity factors supplied by OEM or JDS experience. Equipment usage costs include unit costs ($/hr) for the following elements:

- Maintenance parts;
- Tires;
- Lubricants; and
- Boxes, buckets, and ground engaging tools.

Unit costs for the elements above have been obtained from equipment manufacturer databases and JDS experience. Mobile equipment during the operating period will be Client owned and do not account for any lease, rental, or contractor charges against the equipment.

Equipment mid-life overhauls are included in the sustaining capital costs at 60% the cost of new in addition to the costs below, catered for within sustaining capital costs.

Table 22-8: UG Mobile Equipment Cost Summary

Equipment Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Drill and Blast	0.8	10.7	0.29	11
Drawpoint Operations	2.1	27.3	0.74	27
UG Crush and Convey	0.2	3.2	0.09	3
Shaft Operations	0.1	1.1	0.03	1
Surface Haulage	3.3	43.0	1.16	43
Mine Maintenance	0.1	0.8	0.02	1
Mine General	0.3	4.3	0.12	4
Contingency	0.7	9.0	0.24	9
Total Mining OPEX	**7.6**	**99.5**	**2.7**	**100**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

It has been assumed that lateral development, raise bore, and Alimak development works will be performed by a Contractor with equipment rental fees charged as a monthly indirect cost. Contractor rental equipment costs have been estimated as the cost of new plus 30% for mark-up and financing costs, amortized over a 48-month period. Under this arrangement the development contractors will retain ownership of the equipment at the close of the contract, and budgets have been allocated to procure select units from the development contractor's fleet to remain on site for Owner's use. Buy out costs have been included in the project sustaining capital costs.

It should be noted that these cost assumptions may differ from those which are actually negotiated with development contractors, and the Client reserves the right to procure the equipment in lieu of contractor rental fees.

Mobile equipment requirements and operating costs are located in Table 22-9.

Table 22-9: Mobile Equipment Operating Costs (Excluding Fuel)

Equipment[2]	Peak Requirement	Peak Contractor Supply	Peak Owner Supply	Operating Cost [1] ($/hr)	Contractor Supply Cost ($/mo)
Surface FEL (15 t / 5.4 m^3)	1	1	-	74.77	41,351
Surface Truck (39 t)	4	4	-	43.38	29,436
Surface Light Vehicle / Truck	1	-	1	11.00	-
Surface Loader Crane	1	-	1	9.40	-
Surface Tractor	1	-	1	7.98	-
Surface Telehandler 24T	1	-	1	9.40	-
Surface Telehandler 10T	1	-	1	9.40	-
Surface Warehouse Forklift	1	-	1	9.40	-
LHD (7 t / 2.8 m^3)	1	1	-	69.06	-
LHD (17 t / 7.0 m^3)	4	4	-	134.91	51,611
LHD (21 t / 8 m^3)	3	-	3	156.08	-
Jumbo - 2 Boom	4	4	-	283.91	56,433
Longhole Drill - ITH (Dev)	1	1	-	96.52	19,663
Longhole Drill - ITH (Prod)	5	-	5	96.52	-
Bolter	3	3	-	63.76	47,069
Cable Bolter	2	2	-	91.13	33,596
Shotcrete Sprayer	1	1	-	9.45	1,227
Small Explosives Truck	2	2	-	12.63	10,230
Large Explosives Truck	2	-	2	40.92	-
Transmixer	1	1	-	30.55	9,857
Scissor Lift	3	3	-	8.36	9,857
Fuel/Lube Truck	1	1	-	8.83	9,296
Jackleg/Stoper	1	1	-	8.99	225
Mechanics Truck	2	2	-	11.00	1,121
Electrician Truck	1	1	-	11.00	1,121
Boom Truck	1	1	-	7.98	8,362

Equipment[2]	Peak Requirement	Peak Contractor Supply	Peak Owner Supply	Operating Cost [1] ($/hr)	Contractor Supply Cost ($/mo)
Grader	1	-	1	25.63	-
Mobile Rock Breaker	1	-	1	18.19	-
Telehandler UG	3	-	3	9.40	-
Mobile Conveyor Loader	-	-	-	10.18	-
Supervisor Truck	2	2	-	11.00	1,121
Utility Vehicle	3	3	-	11.00	1,121
Ambulance	1	1	-	11.00	1,121

Notes:
[1]Exclusive of mark-up if supplied by contractor.
[2]Equipment fleet for entire operation. Some of which used only during capital development will not contribute to operating costs
Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

22.2.2.3 UG Mining Consumables

Mining consumable usage rates are built up based on the mine plan quantities for development and production activities. Mining consumables include:

- Drill bits and steel;
- Explosives;
- Ground support;
- Piping;
- Electrical cables;
- Ventilation ducting;
- Hoses and fittings;
- Crusher and conveyor parts;
- Hoist and headframe parts; and
- Maintenance tooling.

Consumable unit costs are based on quotations from local suppliers, many of which already provide KDM with OP consumables. Minor item costs are based on catalog or database values. Ten percent of the base pricing has been added to account for delivery (freight) to site.

Table 22-10: UG Mining Consumables Summary

Consumable Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Drill and Blast	1.7	22.7	0.61	78
Drawpoint Operations	-	-	-	-
UG Crush and Convey	0.1	1.1	0.03	4
Shaft Operations	-	-	-	-
Surface Haulage	-	-	-	-
Mine Maintenance	0.2	2.8	0.07	9
Mine General	-	-	-	-
Contingency	0.2	2.7	0.07	9
Total Mining OPEX	**2.2**	**29.2**	**0.8**	**100**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

22.2.2.4 UG Fuel Consumption

UG mining fuel consumption has been built up based on the required equipment operating hours dictated by the mine plan for development or production-based equipment, and annual allowances for support or fixed infrastructure equipment, based on experience at similar operations. Equipment fuel consumption rates have been sourced from local equipment vendors or the list of CANMET-MMSL approved diesel engines for use in UG mines.

The unit fuel price used in the estimate is $1.20/litre, inclusive of delivery to site.

Table 22-11: UG Fuel Cost Summary

Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Drill and Blast	-	0.2	0.01	-
Drawpoint Operations	1.0	12.5	0.34	26
UG Crush and Convey	-	-	-	-
Shaft Operations	0.1	1.7	0.05	4
Surface Haulage	2.0	26.3	0.71	54
Mine Maintenance	0.1	0.9	0.02	2
Mine General	0.2	2.5	0.07	5
Contingency	0.3	4.4	0.12	9

Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Total Mining OPEX	**3.7**	**48.5**	**1.3**	**100**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

Mobile equipment engine and fuel consumption specifications are listed in Table 22-12. All engines are rated tier 3 as there is not access to ultra-low sulfur diesel fuel in Botswana to support the operation of tier 4 or 5 motors.

Table 22-12: Mobile Equipment Engine and Fuel Consumption

Equipment Description	Engine Make	Engine Model	CANMET Fuel Consumption (l/hr @ 2200RPM)
Surface FEL (15 t / 5.4 m^3)	CAT	C93	52.6
Surface Truck (39 t)	CAT	C18 ACERT	21.0
LHD (7 t / 2.8 m^3)	Volvo	Cat® 3306B DITA	68.6
LHD (17 t / 7.0 m^3)	Volvo	TAD1341VE_369hp	77.7
LHD (21 t / 8 m^3)	Volvo	TAD1344VE_472hp	90.4
Jumbo - 2 Boom	Cummins	QSB4.5_170hp	36.2
Bolter	Detroit Diesel	9043 MU32_148hp	29.2
Cable Bolter	Detroit Diesel	9043 MU32_148hp	33.7
Shotcrete Sprayer	Detroit Diesel	9043 MU32_173hp	34.3
Small Explosives Truck	Deutz	D914 L06_100hp	21.5
Large Explosives Truck	Deutz	D914 L06_100hp	21.5
Transmixer	Deutz	D914 L06_100hp	21.5
Scissor Lift	Deutz	D914 L06_100hp	21.5
Fuel/Lube Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Mechanics Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Electrician Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Boom Truck	Deutz	D914 L06_100hp	21.5
Grader	Deutz	BF6M1013CP_221hp	43.4
Mobile Rock Breaker	Detroit Diesel	9043 MU32_148hp	29.2
Telehandler UG	Deutz	TCD3.6L4	12.6
Supervisor Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Utility Vehicle	Toyota	1106D-E66TA/C6.6_127hp	34.0

Equipment Description	Engine Make	Engine Model	CANMET Fuel Consumption (l/hr @ 2200RPM)
Ambulance	Toyota	1106D-E66TA/C6.6_127hp	34.0

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

22.2.2.5 UG Power Consumption

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level. UG mining power includes the power consumption of the UG crushing circuit, headframe, hoists, and surface compressors.

Electricity unit cost is based on a budgetary rate of $0.0809/kWh.

Table 22-13: UG Power Cost Summary

Operating Costs	Average Annual (US$ M)	Life of Mine (US$ M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Drill and Blast	-	0.4	0.01	1
Drawpoint Operations	-	0.1	-	-
UG Crush and Convey	0.2	2.9	0.08	4
Shaft Operations	2.2	28.3	0.77	38
Surface Haulage	-	-	-	-
Mine Maintenance	-	0.1	-	-
Mine General	2.7	36.0	0.97	48
Contingency	0.5	6.8	0.18	9
Total Mining OPEX	**5.7**	**74.5**	**2.02**	**100%**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG – r2

A power consumption summary is provided in Table 22-14.

Table 22-14: UG Power Consumption

Power Consumptions	Average Annual (MWh)	Life of Mine (MWh)	Unit Cost per tonne Processed (kWh/t)	Weighting (%)
Mobile Equipment	0.6	11	0.0003	1
Ventilation	9.9	178	0.0048	17
Mine Air Cooling	7.0	119	0.0032	11
Shaft and Hoisting	24.9	447	0.0121	43
Crusher and Conveyor	2.7	38	0.0010	4
Dewatering	5.6	101	0.0027	10
Maintenance Facilities	8.2	139	0.0038	13
Misc Other	0.3	5	0.0001	-
Total Load	**59.2**	**1,038**	**0.0281**	**100**

Source: JDS (2023) - LUCKAR14E_FS_OPEX UG - r1

22.2.2.6 Contingency

A 10% contingency has been applied to UG operating costs to account for estimate uncertainties.

22.2.2.7 Mining Cost Metrics

Mine development cost metrics derived from the KDM estimate are summarized below and used to benchmark and validate the mine plan operating costs. Some metrics apply to capital development activities only and have been summarized here for consolidation purposes.

- Longhole Drilling* – US$23/m drilled;
- Drill and Blast - $1.97/t; and
- Crush and Convey - $0.30/t.

*Direct costs of equipment, labour, and materials. Excludes indirects.

22.3 Processing Operating Cost Estimate

The process plant and site infrastructure costs are based on the existing plant yearly operating budget provided by Lucara Botswana and include the following:

- Costs to manage and operate the process plant, audit plant, CRD and FRD facilities and water treatment facilities;
- Site power; and

- Engineering labour for site facilities outside the OP.

Processing costs are not expected to change with the transition to underground mining. Annual operating budgets have been projected over the duration of the UGP without adjustment.

A summary of costs is provided in Table 22-15.

Table 22-15: Processing OPEX

Process Operating Costs	Average Annual	Life of Mine	tonnes Processed	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Process Costs	24.7	493.7	52.2	9.5	82
Other Power Costs	5.3	105.2	52.2	2.0	18
Total	**29.9**	**598.9**	**52.2**	**11.5**	**100**

Source: Lucara (2023) - Karowe FS Model V1.7

22.4 General and Administration Operating Cost Estimate

The site General and Administrative (G&A) costs are based on the existing plant yearly operating budget provided by Lucara Botswana and include the costs associated with the following:

- Site finance and administration;
- Human resources;
- Safety, health and environment;
- Mining and Mineral Resource management; and
- Security.

G&A OPEX in these areas include labour costs, along with all equipment and office supplies, training, fees and permits, and external consultants to support each department as identified by the site.

A summary of costs is provided in Table 22-16.

Table 22-16: G&A OPEX

G&A Operating Costs	Average Annual(1)	Life of Mine	tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
G&A	18.3	365.8	52.2	7.0	100
Total	**18.3**	**365.8**	**52.2**	**7.0**	**100**

Source: Lucara (2023) - Karowe FS Model V1.7

As the operational ramp up of the UG operation coincides with the end of OP mining, outside of the construction period, the overall G&A requirements for the site are not anticipated to change significantly with the inclusion of the UG.

22.5 Cost of Sales and Corporate Operating Cost Estimate

Off-site, in-country corporate costs such as Lucara Botswana management, cost of sales, and direct costs associated with the Clara sales platform have been provided by Lucara. These costs represent costs not directly associated with operating the immediate site, but costs that are still attributable to the Project. The UGP is not anticipated to impact the yearly off-site, in-country costs; as such, the current operational budget provided by Lucara has been extrapolated over the LOM.

A summary of costs is provided in Table 22-17.

Table 22-17: LOM Sales and Corporate Cost

Sales Operating Costs	Average Annual(1)	Life of Mine	tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	M$	M$	Mt	$/t	%
Cost of Sales	4.4	87.9	52.2	1.7	36
Corporate Charges	8.0	159.2	52.2	3.1	64
Total	**12.4**	**247.1**	**52.2**	**4.7**	**100**

Source: Lucara (2023) - Karowe FS Model V1.7

23 ECONOMIC ANALYSIS

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in diamond prices and grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This Technical Report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 and Section 22 of this report (presented in 2023 dollars). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Summary of Results

The summary of the mine plan and payable diamonds produced is outlined in Table 23-1. The summaries provided represent the LOM outputs, which include the remaining OP, current stockpiles and the additional value from the development of the UG.

Table 23-1: Life of Mine (LOM) Summary

Parameter	Unit	Value
Ore Processed	Mt	52.2
Mill Average Annual Production	Mt	2.7
Average Processing Grade	cpht	13.10
Diamonds Recovered	k ct	6,834
Recovery	%	100.0
Initial Capital Cost (inc. Contingency and excluding sunk costs to June 30, 2023)	$M	419

Parameter	Unit	Value
Sustaining Capital Cost	$M	334
Life of Mine Capital	$M	752

Source: Lucara (2023) - Karowe FS Model V1.7

Other economic factors include the following:

- Discount rate of 8%;
- Nominal 2023 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- No escalation of costs or diamond price;
- No inflation;
- Canada corporate (Lucara Diamond Corp.) costs are not included in the economic model results except as noted;
- Lucara Botswana corporate costs are included in all economic results;
- Debt financing costs included;
- Working capital included; and
- The model excludes all sunk costs up to the base date of June 30, 2023.

23.2 Assumptions

Table 23-2 and Table 23-3 outline the diamond prices and exchange rate assumptions used in the economic analysis. The diamond prices have been provided by Lucara and are based on historical information, market assessments and statistical analysis of the anticipated size distribution supported by data sets derived from the existing operations (Section 19).

Table 23-2: Economic Assumptions

Item	Unit	Value
Base Case NPV Discount Rate	%	8
BWP:US$ FX	BWP:US$	12.5
ZAR:US$ FX	ZAR:US$	17

Source: JDS (2023)

Table 23-3: Baseline Diamond Prices

Unit	Unit	2023 FS
North	$/ct	273
Centre	$/ct	392
EM/PK(S)	$/ct	828
M/PK(S)	$/ct	707
Stockpiles	$/ct	574

Source: JDS (2023)

Efforts have been made to provide realistic estimates for diamond prices and exchange rates based historical performance, current sales information and potential future markets. The exchange rates used in the economic model are about 10% lower (less favourable to the project) than the rates as of the effective date of the report (i.e. 13.4 BWP:USD and 18.8 ZAR:USD).

It should be noted that diamond prices and exchange rates are based on many complex factors and there are no reliable long-term predictive tools.

23.3 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate value of the potential mine economics. The completed tax model was prepared by Lucara Botswana and contains the following assumptions:

- Income Tax: Annual tax rate = 70 – 1500/x:
  - Where x is the profitability ratio, given by taxable income as a percentage of gross income;
  - Where the calculated rate shall not be less than the company rate of 22%; and

- Net Losses, incurred in years of high CAPEX expenditures, can be deferred to future years to offset tax liabilities.

- VAT modelled with a three-year delay on refunds; and
- Withholding taxes on foreign consulting services included as a capital cost within the Owner's CAPEX.

Total taxes for the remaining Project LOM are estimated at the amount of $836M.

23.4 Royalties

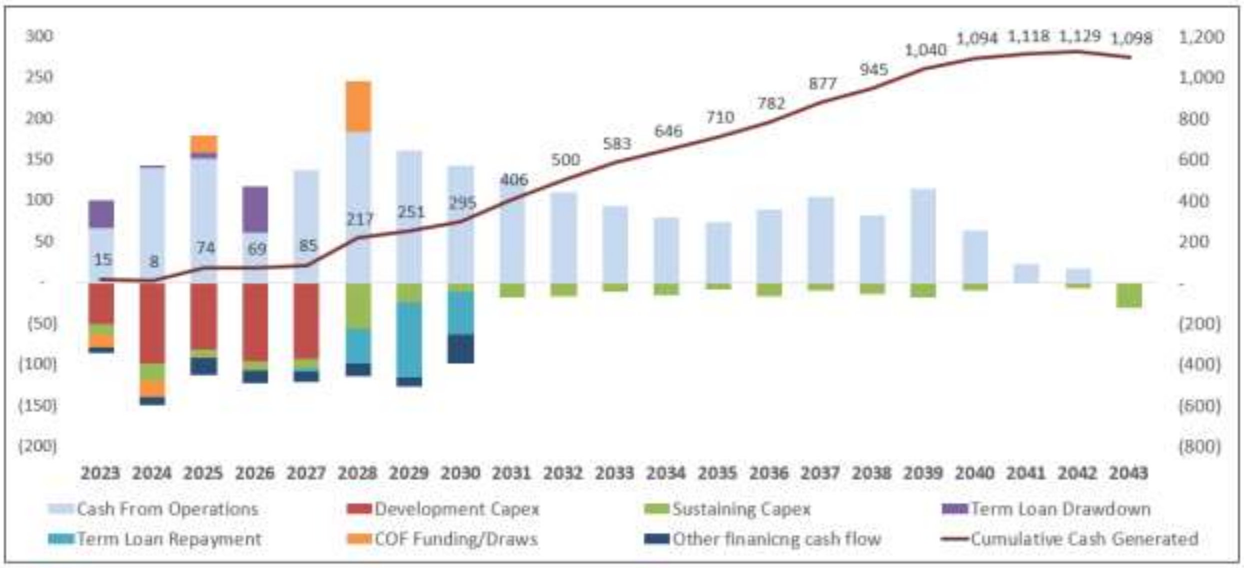
KDM is subject to a royalty payable to the Botswana Government of 10% of all sales. Estimated royalty payments amount to $507M over the remaining LOM.

23.5 Results

The KDM LOM, including the development of the UGP, is economically viable with an after-tax net present value using an 8% discount rate (NPV_8%) of $532[1] M using the diamond prices described in Section 23.2. Figure 23-1 shows the projected KDM cash flows, and Table 23-4 summarizes the economic results for the mine.

[1] NPV reported excludes Canadian corporate costs.

Figure 23-1: After-Tax Cash Flows



Source: Lucara (2023) - Karowe FS Model V1.7

The after-tax break-even average diamond price is approximately $483 /ct or 65% of the base case diamond prices, based on the LOM plan presented herein. This is the weighted average diamond price from each kimberlite type at which the Project NPV_{11}% discount rate is zero.

The life of mine all-in sustaining cost (AISC) is $375/ct. The straight AISC cost is calculated by adding the sales and Botswana corporate, royalty, operating, and capital and closure costs together and dividing by the total payable carats.

The LOM economic model does not calculate a meaningful Internal Rate of Return (IRR), as capital costs are partially offset by operating revenue during the years they are incurred.

Project economic results are reported in Table 23-4 including and excluding any Canadian Corporate Costs. The base case and the sensitivity results are based on the inclusion of Lucara Botswana Corporate costs but not Lucara Diamond Corporate (Canadian corp.) costs.

Table 23-4: Post-Tax Economic Results - LOM Model

Parameter	Unit	After-tax Results
$NPV_{8\%}$ including Canadian corporate costs	US$M	433.1
$NPV_{5\%}$ including Canadian corporate costs	US$M	562.5
$NPV_{8\%}$ excluding Canadian corporate costs	US$M	531.8

Parameter	Unit	After-tax Results
$NPV_{5\%}$ excluding Canadian corporate costs	US$M	684.5

Source: Lucara (2023) - Karowe FS Model V1.7

23.6 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -20% to +20%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the diamond prices were evaluated at a +/- 20% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as diamond prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

Notwithstanding the above noted limitations to the sensitivity analysis, the analysis revealed that the Project is most sensitive to diamond prices and grade. The Project showed the least sensitivity to capital costs. Table 23-5 show the results of the sensitivity tests.

Table 23-5: Sensitivity Results (Post-Tax NPV @ 8%)

Variable	Post-tax $NPV_{8\%}$ (M$)				
	-20% Variance	-10% Variance	Base	+10% Variance	+20% Variance
Diamond Price	252.3	400.1	531.8	672.0	811.3
Mining Cost	556.8	544.3		519.2	506.7
Processing Cost	561.6	546.4		517.1	502.4
All Operating Costs	607.1	568.1		495.6	459.6
Upfront CAPEX	584.6	556.6		509.3	487.0
Sustaining CAPEX	548.1	539.9		523.6	515.5
All capital costs	602.3	565.4		501.2	473.1

Source: Lucara (2023) – Karowe FS Model V1.7

The economic model summary used for the LOM mine is summarized in Table 23-6:.

Table 23-6: LOM Annual Cash Flow

	Year	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Ore Mined																								
Open pit - Ore	North	Ktonnes	11.6	11.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Centre	Ktonnes	587.0	311.0	276.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - EM/PK(S)	Ktonnes	1,272.5	308.4	737.8	226.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - M/PK(S)	Ktonnes	3,612.6	784.1	1,986.2	842.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL	Ktonnes	5,483.7	1,415.0	3,000.0	1,068.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Underground - Ore	Underground: EMPK	Ktonnes	18,567.9	-	-	-	111.9	1,118.6	2,404.5	2,446.9	2,182.3	1,860.2	1,575.1	1,862.6	705.4	986.3	892.4	1,008.8	448.0	959.7	5.2	-	-	-
	Underground: MPK	Ktonnes	18,392.2	-	-	-	3.4	129.4	340.5	290.6	555.2	877.3	1,169.9	874.9	2,032.1	1,751.2	1,852.6	1,728.7	2,289.5	1,777.8	2,719.1	-	-	-
	TOTAL	Ktonnes	36,960.1	-	-	-	115.3	1,248.0	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,724.3	-	-	-
Ore mined	Open Pit	Ktonnes	5,483.7	1,415.0	3,000.0	1,068.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground	Ktonnes	36,960.1	-	-	-	115.3	1,248.0	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,724.3	-	-	-
	Total ore mined	Ktonnes	42,443.7	1,415.0	3,000.0	1,068.7	115.3	1,248.0	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,724.3	-	-	-
Waste mined	Open Pit	Ktonnes	2,549.7	1,385.0	1,081.0	83.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground	Ktonnes	984.3	38.0	111.5	127.9	483.9	161.7	61.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total waste mined	Ktonnes	3,534.0	1,423.0	1,192.5	211.6	483.9	161.7	61.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Tons Moved	Open pit - Ore	Ktonnes	5,483.7	1,415.0	3,000.0	1,068.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Open pit - Waste	Ktonnes	2,549.7	1,385.0	1,081.0	83.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Ore rehandled	Ktonnes	11,218.3	349.9	128.7	1,783.3	2,584.7	1,459.5	7.5	22.5	22.5	15.0	7.5	15.0	15.0	15.0	7.5	15.0	15.0	15.0	20.7	2,700.0	2,018.8	-
	Underground - Ore	Ktonnes	36,960.1	-	-	-	115.3	1,248.0	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,745.0	2,737.5	2,737.5	2,737.5	2,724.3	-	-	-
	Underground - Waste	Ktonnes	984.3	38.0	111.5	127.9	483.9	161.7	61.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL	Ktonnes	57,196.0	3,187.9	4,321.2	3,063.6	3,183.9	2,869.2	2,813.8	2,760.0	2,760.0	2,752.5	2,752.5	2,752.5	2,752.5	2,752.5	2,752.5	2,752.5	2,752.5	2,752.5	2,745.0	2,700.0	2,018.8	-
Ore Grade	Open Pit	cpht	14.65	15.36	13.92	15.77	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground: EMPK	cpht	18.10	-	-	-	15.97	18.58	19.66	19.86	20.00	18.62	17.13	13.73	15.08	12.16	14.44	19.92	21.01	22.64	22.91	-	-	-
	Underground: MPK	cpht	10.17	-	-	-	9.30	9.78	9.62	10.01	9.91	9.62	9.45	8.99	10.02	10.38	10.98	10.97	9.70	10.05	10.58	-	-	-
	Total ore mined	cpht	14.22	15.36	13.92	15.77	15.77	17.67	18.41	18.81	17.95	15.74	13.86	12.21	11.33	11.02	12.11	14.27	11.55	14.46	10.60	-	-	-
Mined Carats	Open Pit	k carats	803.6	217.4	417.7	168.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground: EMPK	k carats	3,361.4	-	-	-	17.9	207.9	472.7	485.9	436.4	346.4	269.9	255.7	106.4	119.9	128.8	200.9	94.1	217.3	1.2	-	-	-
	Underground: MPK	k carats	1,870.5	-	-	-	0.3	12.7	32.7	29.1	55.0	84.4	110.5	78.6	203.7	181.8	203.5	189.6	222.2	178.6	287.7	-	-	-
	Total ore mined	k carats	6,035.5	217.4	417.7	168.6	18.2	220.5	505.5	515.0	491.5	430.8	380.4	334.3	310.1	301.7	332.3	390.6	316.3	395.9	288.8	-	-	-
Stockpile Movement																								
Ore Stockpile	Opening balance	Ktonnes		9,735.5	9,743.3	9,966.7	8,258.8	5,674.1	4,222.1	4,267.1	4,304.6	4,342.1	4,379.6	4,424.6	4,462.1	4,499.6	4,537.1	4,582.1	4,619.6	4,657.1	4,694.6	4,718.8	2,018.8	0.0
	Stockpile added	Ktonnes		357.7	352.1	75.4	-	7.5	52.5	60.0	60.0	52.5	52.5	52.5	52.5	52.5	52.5	52.5	52.5	52.5	45.0	-	-	-
	Stockpile removed	Ktonnes		(349.9)	(128.7)	(1,783.3)	(2,584.7)	(1,459.5)	(7.5)	(22.5)	(22.5)	(15.0)	(7.5)	(15.0)	(15.0)	(15.0)	(7.5)	(15.0)	(15.0)	(15.0)	(20.7)	(2,700.0)	(2,018.8)	-
	Closing balance	Ktonnes		9,743.3	9,966.7	8,258.8	5,674.1	4,222.1	4,267.1	4,304.6	4,342.1	4,379.6	4,424.6	4,462.1	4,499.6	4,537.1	4,582.1	4,619.6	4,657.1	4,694.6	4,718.8	2,018.8	0.0	0.0
Ore Milled																								
Ore Milled	North	Ktonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

	Year	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
	Centre	Ktonnes	322.9	163.8	159.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - EM/PK(S)	Ktonnes	19,504.4	194.5	538.2	226.3	111.9	1,118.6	2,397.0	2,446.9	2,167.3	1,860.2	1,575.1	1,862.6	705.4	986.3	892.4	1,008.8	448.0	959.7	5.2	-	-	-
	South - M/PK(S)	Ktonnes	21,133.7	699.0	1,950.5	767.0	3.4	121.9	295.5	230.6	510.2	824.8	1,117.4	822.4	1,979.6	1,698.7	1,800.1	1,676.2	2,237.0	1,725.3	2,674.1	-	-	-
	Stockpile	Ktonnes	11,218.3	349.9	128.7	1,783.3	2,584.7	1,459.5	7.5	22.5	22.5	15.0	7.5	15.0	15.0	15.0	7.5	15.0	15.0	15.0	20.7	2,700.0	2,018.8	-
	Total ore milled	Ktonnes	52,179.3	1,407.2	2,776.6	2,776.6	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,700.0	2,018.8	-
Milled Grade	North	cpht	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Centre	cpht	17.02	19.52	14.43	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - EM/PK(S)	cpht	18.43	21.93	24.38	28.26	15.97	18.58	19.66	19.86	20.00	18.62	17.13	13.73	15.08	12.16	14.44	19.92	21.01	22.64	22.91	-	-	-
	South - M/PK(S)	cpht	10.23	11.22	9.65	12.40	9.30	9.79	9.59	10.03	10.07	9.65	9.41	9.23	9.85	10.24	10.84	11.16	9.87	9.93	10.48	-	-	-
	Stockpile	cpht	9.12	16.38	17.64	15.08	12.08	6.57	6.98	13.29	16.79	9.87	9.69	9.74	8.73	11.04	9.84	12.12	10.63	9.77	10.52	5.67	4.52	-
	Milled grade	cpht	13.10	14.95	13.15	15.41	12.24	11.69	18.52	18.96	18.09	15.83	13.92	12.34	11.21	10.94	12.02	14.44	11.73	14.45	10.50	5.67	4.52	-
Contained Carat	North	k carats	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Centre	k carats	55.0	32.0	23.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - EM/PK(S)	k carats	3,594.7	42.6	131.2	64.0	17.9	207.9	471.2	485.9	433.4	346.4	269.9	255.7	106.4	119.9	128.8	200.9	94.1	217.3	1.2	-	-	-
	South - M/PK(S)	k carats	2,161.1	78.5	188.3	95.1	0.3	11.9	28.3	23.1	51.4	79.6	105.1	75.9	195.1	173.9	195.1	187.0	220.9	171.4	280.3	-	-	-
	Stockpile	k carats	1,023.1	57.3	22.7	268.9	312.2	95.8	0.5	3.0	3.8	1.5	0.7	1.5	1.3	1.7	0.7	1.8	1.6	1.5	2.2	153.2	91.2	-
	Total contained carat	k carats	6,833.8	210.4	365.2	428.0	330.4	315.6	500.1	512.0	488.6	427.5	375.7	333.1	302.8	295.4	324.6	389.8	316.6	390.1	283.6	153.2	91.2	-
Recovery Rate	North	Ktonnes	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
	Centre	Ktonnes	100%	100%	100%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
	South - EM/PK(S)	Ktonnes	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%	0%	0%
	South - M/PK(S)	Ktonnes	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%	0%	0%
	Stockpile	Ktonnes	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
	Recovery rate	Ktonnes	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
Recovered Carat	North	K Carats	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Centre	K Carats	55.0	32.0	23.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	South - EM/PK(S)	K Carats	3,594.7	42.6	131.2	64.0	17.9	207.9	471.2	485.9	433.4	346.4	269.9	255.7	106.4	119.9	128.8	200.9	94.1	217.3	1.2	-	-	-
	South - M/PK(S)	K Carats	2,161.1	78.5	188.3	95.1	0.3	11.9	28.3	23.1	51.4	79.6	105.1	75.9	195.1	173.9	195.1	187.0	220.9	171.4	280.3	-	-	-
	Stockpile	K Carats	1,023.1	57.3	22.7	268.9	312.2	95.8	0.5	3.0	3.8	1.5	0.7	1.5	1.3	1.7	0.7	1.8	1.6	1.5	2.2	153.2	91.2	-
	Total recovered carats	K Carats	6,833.8	210.4	365.2	428.0	330.4	315.6	500.1	512.0	488.6	427.5	375.7	333.1	302.8	295.4	324.6	389.8	316.6	390.1	283.6	153.2	91.2	-
Revenue																								
Applied Diamond Price	North	$/carat		273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0	273.0
	Centre	$/carat		392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0	392.0
	South - EM/PK(S)	$/carat		828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0	828.0
	South - M/PK(S)	$/carat		707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0	707.0
	SP: Mixed	$/carat		574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0	574.0
Revenue	North	$'000		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Centre	$'000		12,537	9,004	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	South - EM/PK(S)	$'000		35,310	108,648	52,958	14,794	172,111	390,178	402,294	358,864	286,813	223,473	211,728	88,104	99,264	106,668	166,354	77,947	179,900	978	0	0	0
	South - M/PK(S)	$'000		55,467	133,102	67,224	223	8,434	20,038	16,361	36,315	56,275	74,323	53,687	137,912	122,938	137,901	132,227	156,143	121,168	198,147	0	0	0

	Year	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
	Stockpile	$'000		40,981	14,066	146,558	127,927	55,001	370	2,297	3,037	1,046	514	1,033	925	1,171	522	1,285	1,127	1,036	1,543	95,137	52,342	0
	Total	$'000	5,073,732	144,295	264,820	266,740	142,944	235,547	410,586	420,953	398,215	344,134	298,310	266,448	226,942	223,373	245,091	299,867	235,217	302,105	200,668	95,137	52,342	0
Average Diamond Price			742.44																					
Operating Costs																								
Total OPEX	Open Pit Mining Costs	$'000s	72,565	17,855	32,947	21,763	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground Mining Costs	$'000s	413,201	-	-	-	-	4,124	45,847	45,784	46,099	38,833	26,046	25,843	25,817	25,817	25,867	25,815	25,817	25,817	25,678	-	-	-
	Rehandle Costs	$'000s	23,618	524	1,348	3,949	5,199	3,151	-	-	-	-	-	-	-	-	-	-	-	-	-	5,404	4,042	-
	Process Costs	$'000s	493,696	15,325	25,957	25,684	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	25,668	16,042	-
	Other Power Costs	$'000s	105,182	2,921	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	5,491	3,432	-
	G&A	$'000s	365,771	10,888	19,971	19,655	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	19,627	14,720	6,133	-
	Cost of Sales	$'000s	87,886	1,908	3,950	4,118	4,332	4,523	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	4,722	2,951	-
	Corporate Charges (Botswana)	$'000s	159,191	3,408	7,101	7,456	7,828	8,186	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	8,561	5,351	-
	TOTAL	$'000s	1,721,109	52,829	96,764	88,115	68,145	70,770	109,915	109,852	110,168	102,901	90,114	89,911	89,885	89,885	89,936	89,884	89,885	89,885	89,747	64,566	37,952	-
Unit costs	Open Pit Mining Costs	$/tonne mined	9.03	6.38	8.07	18.88	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Underground Mining Costs	$/tonne mined	10.89	-	-	-	-	2.93	16.34	16.72	16.84	14.19	9.49	9.44	9.43	9.43	9.42	9.43	9.43	9.43	9.43	-	-	-
	Rehandle Costs	$/tonne reclaimed	2.11	1.50	10.47	2.21	2.01	2.16	-	-	-	-	-	-	-	-	-	-	-	-	-	2.00	2.00	-
	Process Costs	$/tonne processed	9.46	10.89	9.35	9.25	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	9.51	7.95	-
	Other Power Costs	$/tonne processed	2.02	2.08	1.98	1.98	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	2.03	1.70	-
	G&A	$/tonne processed	7.01	7.74	7.19	7.08	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	7.27	5.45	3.04	-
	Cost of Sales	$/tonne processed	1.68	1.36	1.42	1.48	1.60	1.68	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.46	-
	Corporate Charges (Botswana)	$/tonne processed	3.05	2.42	2.56	2.69	2.90	3.03	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	3.17	2.65	-
	All in cost	$/tonne processed	32.98	37.54	34.85	31.73	25.24	26.21	40.71	40.69	40.80	38.11	33.38	33.30	33.29	33.29	33.31	33.29	33.29	33.29	33.24	23.91	18.80	-
Capital Costs																								
Upfront CAPEX	1000 - MINING	$'000	253,098	30,914	61,923	45,124	60,512	54,625	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	2000 - SITE DEVELOPMENT	$'000	13,415	597	1,206	6,039	1,016	4,556	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	3000 - PROCESS PLANT	$'000	132	-	132	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	4000 - TAILINGS	$'000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	5000 - ON-SITE INFRASTRUCTURE	$'000	5,073	1,506	2,314	799	453	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	6000 - BUILDINGS and FACILITIES	$'000	3,099	271	668	701	717	742	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

	Year	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
	7000 - OFF-SITE INFRASTRUCTURE	$'000	371	266	40	40	26	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	8000 - PROJECT INDIRECTS	$'000	21,724	2,973	4,346	5,211	5,042	4,152	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	9000 - OWNER COSTS	$'000	89,900	10,651	20,098	17,358	20,150	21,642	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Initial CAPEX	$'000	386,811	47,178	90,727	75,272	87,916	85,718	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	Sustaining - Mining CAPEX	$'000	132,812	-	-	-	-	3,081	46,517	13,752	5,564	12,003	5,258	5,445	10,013	3,922	4,831	4,401	9,051	4,292	4,681	-	-	-
	Sustaining - Site General	$'000	95,540	7,101	14,421	6,862	8,432	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	4,195	-	-	-
	Sustaining - Exploration (Regional)	$'000	833	833	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sustaining - Tailings	$'000	42,764	4,158	6,720	1,376	2,752	2,934	-	4,188	-	-	5,504	-	-	-	6,737	-	-	8,334	61	-	-	-
	Sustaining - Closure	$'000	34,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5,667	28,333
	Total Sustaining CAPEX	$'000	305,949	12,092	21,141	8,238	11,184	10,210	50,711	22,134	9,759	16,198	14,957	9,640	14,208	8,117	15,762	8,596	13,246	16,821	8,936	-	5,667	28,333
Contingencies	Upfront CAPEX Contingencies	$'000	31,938	3,895	7,491	6,215	7,259	7,077	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sustaining CAPEX Contingencies	$'000	27,624	-	-	1,130	1,165	1,021	5,071	2,213	976	1,620	1,496	964	1,421	812	1,576	860	1,325	1,682	894	-	567	2,833
	Total Contingencies	$'000	59,562	3,895	7,491	7,345	8,424	8,098	5,071	2,213	976	1,620	1,496	964	1,421	812	1,576	860	1,325	1,682	894	-	567	2,833
Annual Cash Flow Statement				6+6	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan	Plan
Revenue	Total Revenue	$M	5,073.7	144.3	264.8	266.7	142.9	235.5	410.6	421.0	398.2	344.1	298.3	266.4	226.9	223.4	245.1	299.9	235.2	302.1	200.7	95.1	52.3	0.0
	Royalties	$M	(507.4)	(14.4)	(26.5)	(26.7)	(14.3)	(23.6)	(41.1)	(42.1)	(39.8)	(34.4)	(29.8)	(26.6)	(22.7)	(22.3)	(24.5)	(30.0)	(23.5)	(30.2)	(20.1)	(9.5)	(5.2)	0.0
	Other revenue deductions	$M	(247.1)	(5.3)	(11.1)	(11.6)	(12.2)	(12.7)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(13.3)	(8.3)	0.0
	Net Revenue	**$M**	**4,319.3**	**124.5**	**227.3**	**228.5**	**116.5**	**199.3**	**356.2**	**365.6**	**345.1**	**296.4**	**255.2**	**226.5**	**191.0**	**187.8**	**207.3**	**256.6**	**198.4**	**258.6**	**167.3**	**72.3**	**38.8**	**0.0**
Operating Costs	Operating Costs	$M	(1,474.0)	(47.5)	(85.7)	(76.5)	(56.0)	(58.1)	(96.6)	(96.6)	(96.9)	(89.6)	(76.8)	(76.6)	(76.6)	(76.6)	(76.7)	(76.6)	(76.6)	(76.6)	(76.5)	(51.3)	(29.6)	0.0
	Corporate Income Tax	$M	(836.2)	0.0	0.0	(10.3)	0.0	0.0	(74.2)	(108.1)	(106.5)	(80.7)	(68.6)	(57.5)	(35.1)	(38.0)	(42.6)	(74.4)	(39.8)	(69.1)	(26.6)	(4.6)	0.0	0.0
	Working Capital and Others	$M	13.6	(10.3)	(2.4)	10.0	0.7	(4.2)	(1.7)	0.1	1.2	2.7	0.8	0.8	(0.5)	0.3	0.5	(0.5)	(0.1)	0.9	(0.0)	6.7	8.4	0.0
	Total Cash Operating Costs	**$M**	**(2,296.6)**	**(57.8)**	**(88.1)**	**(76.8)**	**(55.3)**	**(62.2)**	**(172.5)**	**(204.5)**	**(202.2)**	**(167.6)**	**(144.6)**	**(133.4)**	**(112.2)**	**(114.3)**	**(118.7)**	**(151.5)**	**(116.5)**	**(144.8)**	**(103.1)**	**(49.2)**	**(21.2)**	**0.0**
Cash Flow after Operations		**$M**	**2,022.7**	**66.7**	**139.2**	**151.7**	**61.2**	**137.0**	**183.7**	**161.0**	**142.9**	**128.8**	**110.6**	**93.2**	**78.8**	**73.4**	**88.6**	**105.1**	**81.9**	**113.8**	**64.2**	**23.1**	**17.6**	**-**
Capital Costs	Development Capital - Costs	$M	(386.8)	(47.2)	(90.7)	(75.3)	(87.9)	(85.7)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Development Capital - Contingencies	$M	(31.9)	(3.9)	(7.5)	(6.2)	(7.3)	(7.1)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Sustaining Capital Costs (Incl. Contingency)	$M	(333.6)	(12.1)	(21.1)	(9.4)	(12.3)	(11.2)	(55.8)	(24.3)	(10.7)	(17.8)	(16.5)	(10.6)	(15.6)	(8.9)	(17.3)	(9.5)	(14.6)	(18.5)	(9.8)	0.0	(6.2)	(31.2)
	Total Capital Costs	$M	**(752.3)**	**(63.2)**	**(119.4)**	**(90.9)**	**(107.5)**	**(104.0)**	**(55.8)**	**(24.3)**	**(10.7)**	**(17.8)**	**(16.5)**	**(10.6)**	**(15.6)**	**(8.9)**	**(17.3)**	**(9.5)**	**(14.6)**	**(18.5)**	**(9.8)**	**0.0**	**(6.2)**	**(31.2)**

	Year	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Cash Flow after CAPEX		**$M**	**1,270.4**	**3.6**	**19.8**	**60.9**	**(46.3)**	**33.0**	**127.9**	**136.7**	**132.2**	**111.0**	**94.1**	**82.6**	**63.2**	**64.5**	**71.2**	**95.6**	**67.4**	**95.3**	**54.4**	**23.1**	**11.4**	**(31.2)**
PF Term Loan	Financing costs	$M	(94.1)	(6.6)	(12.2)	(11.9)	(14.0)	(17.1)	(17.1)	(12.2)	(3.2)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Debt drawdown	$M	100.0	33.5	3.0	7.1	56.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Debt service	$M	(190.0)	0.0	0.0	0.0	0.0	(4.4)	(42.4)	(90.5)	(52.7)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Total PF Term Loan	**$M**	**(184.1)**	**26.9**	**(9.2)**	**(4.8)**	**42.4**	**(21.5)**	**(59.4)**	**(102.7)**	**(55.9)**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
WC Facility	Financing costs	$M	(15.3)	(0.9)	(1.7)	(1.7)	(1.5)	(1.9)	(2.5)	(2.8)	(2.3)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	WC Facility utilisation	$M	21.6	(13.1)	12.9	1.3	5.5	6.1	5.6	2.6	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-
	WC Facility repayment	$M	(56.6)	(1.9)	(8.9)	(9.7)	(4.7)	0.0	(1.4)	0.0	(30.0)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Total WC Facility	**$M**	**(50.3)**	**(15.9)**	**2.4**	**(10.0)**	**(0.7)**	**4.2**	**1.7**	**(0.1)**	**(31.7)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Equity	Equity	$M	16.1	(13.7)	0.0	0.0	4.6	25.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Equity from Cash balance	$M	(1.1)	28.7	0.0	0.0	(4.6)	(25.2)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	COF drawing/funding	$M	46.7	(15.0)	(20.0)	20.0	0.0	0.0	61.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Total Equity	**$M**	**61.7**	**0.0**	**(20.0)**	**20.0**	**-**	**-**	**61.7**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Net Financing Cash Flows	**$M**	**(172.7)**	**11.0**	**(26.8)**	**5.1**	**41.7**	**(17.3)**	**3.9**	**(102.9)**	**(87.6)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cash Flow after Financing		**$M**	**1,097.7**	**14.6**	**(7.0)**	**66.0**	**(4.6)**	**15.7**	**131.9**	**33.8**	**44.6**	**111.0**	**94.1**	**82.6**	**63.2**	**64.5**	**71.2**	**95.6**	**67.4**	**95.3**	**54.4**	**23.1**	**11.4**	**(31.2)**

Source: Lucara (2023) - Karowe FS Model V1.7

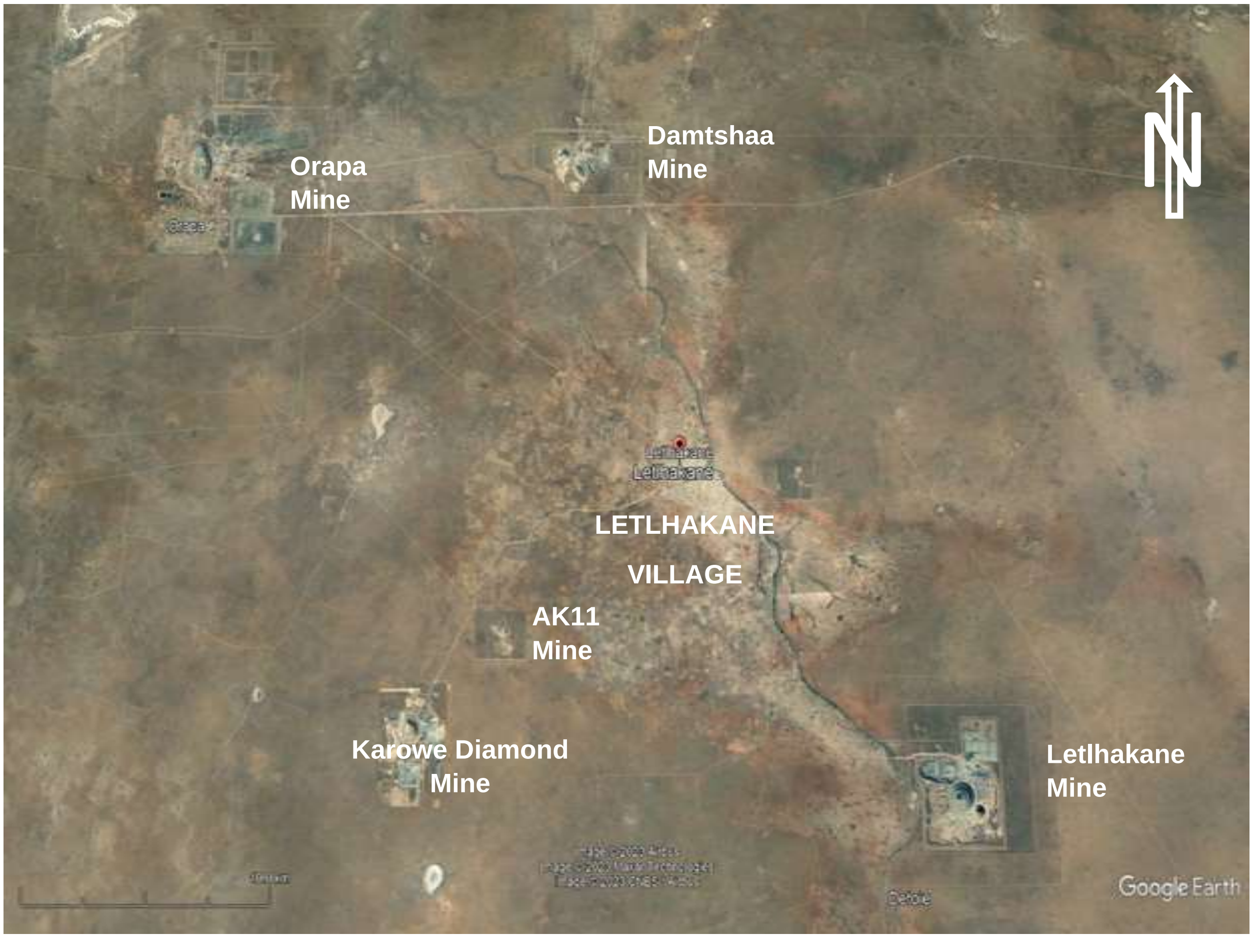
24 ADJACENT PROPERTIES

There are several kimberlite pipes identified within a 50 km radius of KDM as well as several operating and past-producing diamond mines. Debswana Diamond Company, an even joint venture between De Beers and the Government of Botswana owns three mines in the area, the largest and by far the most significant is the Orapa Mine. Anglo American owns 85% of the De Beers Group. The other two Debswana mines, Damtshaa and Letlhakane, have been placed on care and maintenance and remain dormant at this time. A summary of the Debswana owned mines can be found in Table 24-1.

None of the local mines have an impact on the KDM operation except for the Orapa Mine which has an agreement with KDM to take excess water from KDM. Orapa uses water from well fields for its processing facility and infrastructure needs and has committed to take any excess water KDM produces to reduce its reliance on well water. The pumping of water from the mine to Orapa gives KDM flexibility in its disposal of excess water.

Figure 24-1 shows the location of local producing and past-producing mines.

Figure 24-1: Locations of Major Diamond Mines Near KDM



Source: Google Earth

Table 24-1: Summary Information for the nearby Debswana-owned Orapa, Letlhakane and Damtshaa Mines[2]

Parameter	Unit	Orapa	Letlhakane	Damtshaa		
Mining Method		OP	TMR & ORT*	OP	TMR & ORP	OP
M+I Resource Tonnes**	Mt	280	189	22		25
M+I Resource Grade**	cpht	97	67	32		22
Inf Resource Tonnes**	Mt	75	-	19	49	27
Inf Resource Grade**	cpht	86	-	32	27	24
Reserve Tonnes	Mt	91			27	-
Reserve Grade	cpht	103			19	-
Operating life	years	15			21	
Status		Operating	Project	Care and maintenance		Care and maintenance

Notes:
The qualified person has been unable to verify the information contained in this table and that the information contained in the table is not necessarily indicative of the mineralization on the KDM property that is the subject of the technical report.
*TMR = Tailings Mineral Resources, ORT – Old Recovery Tailings
** Resources are reported as additional to Reserves

Source: Anglo American Ore Reserves and Mineral Resources Report (2022)

[2]The QP has been unable to verify the information contained in this table and that the information contained in this table is not necessarily indicative of the mineralization on the KDM property that is the subject of the technical report.

25 OTHER RELEVANT DATA AND INFORMATION

25.1 Project Execution Plan

25.1.1 Introduction

The KDM Project Execution Plan (PEP) describes the project development strategies that were considered for the FS capital cost estimate and project schedule. The PEP is meant to provide the future framework for organizing the engineering, procurement, and construction. The Execution Plan also serves as a guide in:

- Promoting safety in design, construction, and operations in order to succeed;
- Negotiating contracts with suppliers, contractors, and engineers with proven track records in the region; and
- Planning the project execution in a way that allows the project to leverage the existing site workforce and maximizes local labour as much as possible when external contractors are required.

Although the Execution Plan provides guidance for executing the Project, continuous planning will evaluate alternate execution strategies and other opportunities that add value overall. This may include items such as variations to portions of the execution strategy or inclusion of Owner resources for smaller scopes of work.

25.1.2 Project Development Schedule

The overall development period for the Project is estimated to be approximately eight years from the start of detailed engineering (Jan 2020) to reaching 70% production capacity (Nov 2027).

As of the reporting date of this Feasibility Study update, the development schedule is underway, and the following critical path remains to reach project completion:

- Shaft sinking, including station development between shafts;
- Shaft equipping, including procurement and construction of permanent surface winding plant;
- Skip loading infrastructure and fine ore bin construction;
- Removal of V/S headgear and installation of permanent ventilation fans;
- Installation of primary dewatering systems;
- UG excavation of drifts and raises towards the South Lobe;

- Installation of crushing and conveying infrastructure; and
- Drawbell drill and blast.

25.1.3 Development Milestones

The following development milestones apply to the current project schedule:

2024

- P/S will reach the main extraction level of 310 masl;
- V/S will reach the top of the fine ore bin level of 335 masl;
- Shaft stations 670 and 470 will be fully excavated, and 335 partially excavated; and
- Permanent Auxiliary and Cage winders will be delivered to site and installation started. Cage winder building will be erected.

2025

- P/S will reach shaft bottom level of 245 masl;
- Permanent Cage and Auxiliary winders will be installed and commissioned for use in Shaft equipping;
- P/S will be fully equipped with internal steelwork and loading pocket infrastructure;
- V/S will reach shaft bottom level of 285 masl;
- Shaft Stations 335, 310, and 285 will be excavated;
- All but 285 level Shaft stations will be equipped with permanent infrastructure; and
- Fine Ore Bin #1 will be excavated and connected via rock pass system to 470 station.

2026

- P/S Headgear will be changed over, and permanent skips, cages, and bins installed. Shaft will be fully commissioned and handed over to Client;
- Fine Ore Bin #2 will be excavated and both bins equipped with loading and discharging infrastructure;
- Skip loading conveyor and associated infrastructure will be installed and commissioned;
- Lateral Development, Infrastructure installation, and Raise bore contractors deployed UG;

- Vertical Dams and Flood Drift excavated and Main Pump Station installed;
- 310 Extraction Level excavated. First Ore achieved;
- Crusher and conveyor chambers excavated and construction commenced;
- Ramp to Shaft bottom excavated and shaft bottom infrastructure installed;
- V/S and Headgear stripped, Main Fans installed and commissioned; and
- Primary ventilation circuit established UG and Bulk Air cooler commissioned.

2027

- 340 undercut station developed and draw bells drilled and blasted;
- 380 production horizon developed and stoping commenced;
- 470 production horizon developed;
- 580 ramp to 580 production horizon developed; and
- Production ramp up to 75% throughput achieved.

A level 1 schedule as shown in Table 25-1 illustrates the UGP development.

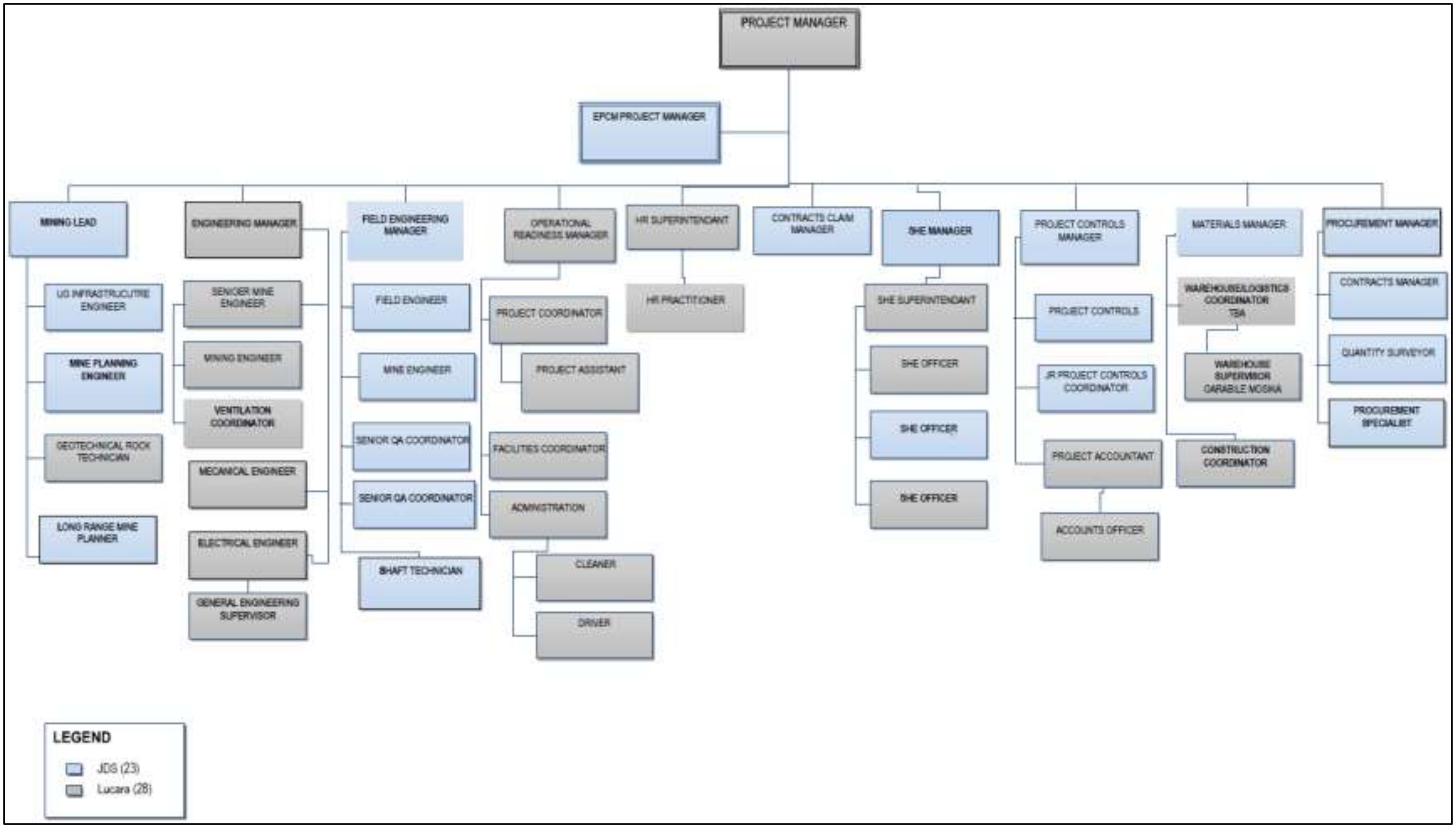
Table 25-1: KDM UG Development Schedule

	2023	2024		2025		2026		2027	
	H2	H1	H2	H1	H2	H1	H2	H1	H2
Production Shaft									
Sink									
Equip									
Ventilation Shaft									
Sink									
UG Construction									
Equip									
UG Development									
- LEVEL: 245									
- LEVEL: 285									
- LEVEL: 310									
- LEVEL: 335									
- LEVEL: 340									
- LEVEL: 380									
- LEVEL: 470									
- LEVEL: 580									
- LEVEL: 670									
UG Infrastructure									
Pump Station									
Workshop									
Crusher									
Conveyor									
Draw bells									
UG Production									
- LEVEL: 380									
- LEVEL: 470									
- LEVEL: 580									
- LEVEL: 670									

25.1.4 Project Management

The Project Management Team (PM Team) is an integrated team including the Owner's personnel, the EPCM contractor, and various engineering contractors. The PM Team oversees and directs all engineering, procurement, and construction activities for the Project. Figure 25-1 presents the current project organization chart for both the engineering and construction phases of the Project.

Figure 25-1: Organizational Structure



Source: JDS (2023)

25.1.5 Engineering

The general engineering execution strategy for the Project will be to utilize multiple engineering firms with specialized knowledge of their assigned scope. Coordination of engineering interfaces and overall management of engineering schedule and deliverables will be the responsibility of the EPCM project manager or infrastructure and mining leads. The following major engineering contract packages have been identified for the Project and remain to be completed:

- EPCM Services;
- Shaft Design and Procurement;
- UG Electrical Engineering;
- UG Dewatering Engineering;
- Surface Dewatering Engineering;
- UG Crushing and Conveying Engineering;
- UG Fire Protection Engineering; and
- UG General Infrastructure Engineering.

25.1.6 Construction

During the construction Phase, the Project Manager (or their designate) will be responsible for the development and construction areas. The designated EPCM Construction Manager and Client Engineering Manager will closely coordinate site activities to maintain project efficiency. The main objectives of the construction execution strategy will include:

- Execute all activities with a goal of zero harm to people, assets, the environment, or reputation;
- Strive to eliminate process, operational and maintenance safety hazards;
- Meet or exceed environmental regulatory and permit requirements;
- Cultivate an atmosphere of positive social impact in the surrounding communities;
- Maximize the involvement of the existing site workforce;
- Utilize local labour as much as possible;
- Identify and remove barriers that affect project progress; and

- Recognize, identify and communicate outstanding achievements during construction and commissioning of the Project.

26 INTERPRETATIONS AND CONCLUSIONS

26.1 External Risks

Almost every mining project has a large variety of risks that can be controlled at the mine site (internal risks) or are beyond the control of the site (external risks).

External risks to KDM and UGP have not been described in detail in this report include, but are not limited to:

- General global and local geopolitical, financial and economic conditions including inflation, recession, etc.;
- Wars and military actions;
- Economic sanctions;
- Global supply chain challenges and disruptions;
- The effects of global pandemics;
- Commodity pricing and availability;
- Acts of the governments that impact Lucara's business;
- Future market prices for diamonds;
- Commercial success of the Clara platform;
- Risks associated with climate change including the impact of extreme weather events on mining operations;
- Fluctuations in interest rates, foreign currency exchange rates and tax rates;
- Lucara's ability to protect its intellectual property, including in foreign jurisdictions;
- Risks associated with the production and increased consumer demand for synthetic gem-quality diamonds;

Further descriptions of the external risks can be seen in 2022 Lucara Diamond Corp. Annual Information Form found on the Lucara https://lucaradiamond.com/ or SEDAR+ at www.sedarplus.ca websites.

The internal risks highlighted in the remainder of this section are general in nature and are further defined in Lucara risk registers.

26.2 Internal Risks

The internal risks highlighted in the remainder of this section are a combination of general internal risks and detailed technical and other risks described and identified during the build-up of the Project risk registers.

The UGP risk registers have been developed throughout the Project life and are living documents that are regularly re-assessed as the project matures and risks are removed and/or additional risks are recognized.

Risk identification within the registers was based on direct interviews and inputs from the disciplines leads: geotechnical, hydrogeology, mining, shafts sinking and CRD/FRD management. The Project Execution Risk Register is presented in Table 26-1.

26.2.1 Risk Registers

Lucara maintains risk registers for the Project managed with CURA risk management software:

- Shafts Sinking Safety Risks Register – Register active, monitored and updated bi-monthly;
- Water Risks Register – Register active, monitored and updated monthly;
- Lateral Development Risks Register – Register active, monitored and updated monthly; and
- Shafts Permanent Configuration Safety Risks Register – Register currently being implemented.

The management of Risks under the UGP is performed in compliance with Lucara Botswana Enterprise Risk Management (ERM) Framework, and Lucara Botswana Enterprise Risk Management (ERM) Policy.

Table 26-1: FS Risks Register Main Project Risks (not necessarily in order of significance)

Risk Statement	Risk Category	Description/Cause/Consequence	Mitigation
Risks associated with financing requirements	Administration	Lucara expects to use a combination of cash flow from operations and external financing for this expansion project and as such a substantial portion of Lucara's revenues and cash flows are committed to the UGP at KDM. To the extent that Lucara does not generate sufficient revenues and operating cash flow to satisfy its obligations in connection with the UGP and its debt obligations, or the capital cost of the Project increases, it will require additional capital. Lucara may not be successful in locating suitable additional or alternate financing when required or, if available, may incur substantial fees and costs and the terms of such financing might not be favourable to Lucara. A failure to raise capital when needed could have a material adverse effect on Lucara's business, financial condition, and results of operations.	Appropriate operational and Project controls are established as well as continual, focused identification and pursuit of optimizations and improvements by the site teams. Ongoing strong connection and communication with shareholders and Lenders.
Ability to attract and retain skilled employees and contractors both local and international	Administration	Competitive market is expected locally by other mining projects in the vicinity of KDM; expected high local demand for various construction support services (transport, fuel supply, customs services, aggregates, food supply, etc.) and construction equipment. Delays or rejection of key expat work permits.	The UGP to date has had an excellent record of attracting local and international personnel to drive the project. Exceptional, experienced people are always in high demand, but fair, industry-standard compensation packages are offered by Lucara and its subcontractors resulting in the ability to retain the necessary skilled people. Close communication with government on work permit needs.
Inaccuracies associated with Mineral Resource and reserve estimation	Geology/ Engineering	The geological model shape, kimberlite types, diamond grade, size and quality estimates are based only on drillholes and samples making the Mineral Resource and reserve estimates and ultimately mining shapes only an estimate.	Lateral mine development in and around the kimberlite will define the pipe shape on multiple levels well in advance of stoping and inform final detailed designs. Short drillholes to further define certain parts of the kimberlite may be warranted depending on how the actual shapes compare to modelled shapes early on.
UGP construction schedule delay	Cost/schedule	If the UGP is delayed for any number of reasons (see the risk factor: "Capital Costs Related to the UGP" below): • The mine revenue will rely on processing OP stockpiled material which is at lesser value than the planned UG ore value; • The overall cost of the UGP could materially increase due to an extension of indirect costs; and • The combination of above could require additional project financing of which the risks are described above.	The project has an established project schedule that is reviewed continuously, and delay mitigation efforts are continually enacted as needed. Value Engineering is performed regularly to optimise plans and schedules. The Project is engaging with leaders of technical disciplines (i.e., shaft grouting) to ensure works can be completed safely and on time.
Capacity and availability of contractors and suppliers to provide construction support services and equipment.	Cost/schedule	A competitive market is expected locally due to other mining projects in the vicinity of KDM; expected high local demand for various construction support services (transport, fuel supply, customs services, aggregates, food supply, etc.) and construction equipment. Works planned to develop the UGP are not common to Botswana and may rely heavily on skilled trades from outside the country. A large portion of project works have been budgeted under the assumption that a local workforce will be trained to take over infrastructure which is completed by contractors. A failure to supply this workforce will cause extensions to contractor works and higher project costs.	Commitment to early procurement, logistics planning, and appropriate compensation are being implemented. Operational Readiness team is in place and plans for Client take-over are underway.
Dilution from host rock	Technical Risk - Geotechnical	Sudden or excessive failure of waste host rock could cause major inflow the stope excavation increasing planned dilution and, in the event of a sudden massive failure, potentially causing an air blast through tunnels and shafts.	The stoping sequence is designed to leave the stope almost full of blasted ore providing support to the host rock walls and reducing the amount of void space within the stope. Draw control from the stopes will be imperative to maintain and will be a priority for mine management during the phase of drilling, blasting and mucking of the stope until the entire orebody is broken. After the entire pipe is blasted, the ore can be drawn as quickly as the mine infrastructure permits so the time the stope remains open, and the host rock left unsupported is minimized.
Brow sloughing and large fragmentation / oversize ore material	Technical Risk - Mining	Long drillholes (up to 100m) over widely spaced drilling horizon can lead to hole deviation and less effective blasting and rock breakage potentially generating oversize material that may affect draw control and block drawpoints.	The primary mitigation for maintaining the effectiveness of long holes is to survey them all and do re-drills on holes that exceed deviation limits. This control has been conducted effectively at many other mining operations and is a standard practice in long hole stopes. The use of the planned in-the-hole (ITH) hammer drills greatly increases drilling accuracy over long holes and are well-established in the industry. Design flexibility allows reduction of length of drillholes and the addition of drilling sublevels if needed.

Risk Statement	Risk Category	Description/Cause/Consequence	Mitigation
			Plans for oversize management at the drawpoints are in place with provisions for mobile breakers and secondary breakage drills to clear blocked drawpoints.
Presence of gases in the UG mine	Technical Risk - Mining	Carbonaceous shales and kimberlite have both been found to be sources of methane gas. Methane is a common explosive gas found in sedimentary rock mine, particularly coal mines. Levels of methane gas emissions can trigger threshold for mine classification as gaseous or fiery mine under applicable regulations, with consequences for equipment specification. Mine equipment has not been specified as flameproof (suitable for a fiery mine), nor is flameproof equipment available in the sizes selected for the mine plan.	Continued gas emission monitoring for all potentially concerning gases (H_2, N_2, NO_x, CO, CO_2, CH_4, etc.) will be done as per the Ventilation Management Plan. Ventilation systems have been designed to dilute potential gas emissions below dangerous levels and the continued monitoring and adjusting of the ventilation system will be done to provide as much fresh air as practical for new development headings. The drilling of a large number of long blast holes will likely provide an effective method of bleeding off any pockets of gas that may be encountered on the kimberlite prior to stope blasting.
Sill pillar failure	Technical Risk - Mining	Sill pillars have been specified through geomechanical modeling informed by drillhole geotechnical, hydrogeological, and geological data and are expected to remain stable at the specified thickness throughout the planned mining sequence, and according to the modelled geotechnical conditions. Geological features (not included in the model) may allow the formation of large blocks, which could topple into the excavation. This risk is increased during the extraction of the perimeter stopes in the sill, when blocks will be bounded by free faces, the weaker, jointed contact zone, and possible faults. A compromised pillar may require additional work-around development, investigative drilling, and geotechnical evaluations at the cost of reduced production and increased project cost. Pillars which cannot be mitigated may cause resource sterilization and premature project closure. Sudden or unplanned block failures could result in equipment and personnel falling into the stope.	The kimberlite has high rock strength, high resistance to weathering and relatively few joints. . Pit sump will be actively dewatered until crown pillar is wrecked to prevent water to stand and leach into the crown pillar. The pyramidal mining sequence creates a compressive arch, which will clamp blocks, until the extraction of perimeter stopes in the sill, when this effect is reduced. Carefully planned drilling and blasting operations will have a large impact on mitigating potential large failures. Structural and geotechnical mapping of the drill horizon development, and the contact zone, followed by the preparation of a structural model, will assist in the evaluation of the potential for block failure. The stope back shape, rock condition and broken muckpile level will be continuously monitored with geotechnical devices like extensometers and cavity monitoring systems. Mass blasting of perimeter stopes may be required to ensure safety of personnel. Perimeter drives in the host rock will mitigate the risk associated with the perimeter stopes in the sill. This will allow more escape routes and the perimeter stopes could be blasted through additional blastholes drilled from the host rock perimeter drive. There will be an additional cost due to the additional waste development, and recovery and fragmentation may be compromised
Crown pillar failure	Technical Risk - Mining	The crown pillar has been tested through numerical modelling, informed by drillhole geotechnical, hydrogeological, and geological data and are expected to remain stable at the specified thickness throughout the planned mining sequence, and according to the modelled geotechnical conditions. The current crown pillar extraction sequence is complex, incorporating a mass blast, which is necessary for the safety. There is a risk that there are unusual geotechnical conditions, or the rock mass response is different from that anticipated. If the risk of crown pillar failure cannot be mitigated, this may cause resource sterilization. Sudden or unplanned failures could result in equipment and personnel falling into the stope.	During mining, the rock mass response will be monitored and assessed. As more information is gathered on the geotechnical characteristics and behaviour, it will be necessary to update the model to take this into consideration. The stability of the crown pillar should be re-assessed and re-designed if required.
Build-up of water in the stope during mechanical failure / downtime of material handling equipment and risk of flooding in the extraction area following re-start of extraction	Technical risk - Mining	During downtime of material handling equipment, the stope inventory must be kept moving to maintain mixing of dry / wetter materials and prevent potential accumulation of water. Minimum draw shall continue even if no material handling is taking place.	Design includes availability of temporary storage of ore to achieve a minimum draw of six buckets per day, for four days, thus allowing for maintaining movement and mixing of the muck pile.

Risk Statement	Risk Category	Description/Cause/Consequence	Mitigation
Confidence in the mining method – "bottom-up" Long Hole Shrinkage (LHS)	Technical risk - Mining / Geotechnical	The "bottom-up" LHS mining method is unprecedented in diamond mining to the scale being considered for the present project. Absence of other similar applications at the scale of the present project creates technical uncertainties. Available benchmarked operations were terminated prematurely for geotechnical complications during the mining sequence.	The proposed mining method takes advantage of and benefits from the unique KDM kimberlite features such as high-density and high-strength, low weathering potential, few geological discontinuities, and cylindrical shape of the KDM kimberlite. The mining method is supported by strong back-up of data from extensive drilling and geotechnical modelling. In depth discussions with benchmarked operations to extract lessons learned for application to the KDM mine design and plan. Flexible mine design back-bone infrastructure allows for a change to mine method or reduced level spacing should mining not go to plan.
Process Plant Recoveries	Technical Risk - Processing	Negative changes to mineral processing assumptions could lead to reduced diamond recovery, increased processing costs, and/or changes to the processing circuit design.	Additional sampling and testing are recommended. The testwork should include variability composites (VC) of discrete drill core intervals that spatially represent the areas to be mined.
Uncertainty in groundwater Inflows to be pumped higher than predicted	Water Risks	Based on the September 21, 2023 update of the 3-D hydrogeological model. As with all models, there is an inherent level of uncertainty in the 3-D hydrogeological model. The criticality of this uncertainty lies in the case where the estimated inflows would be greater than predicted, thus exceeding the capacity of the infrastructures constructed for managing the UGP volumes of water in the UG mine as well as at the surface where these volumes are to be pumped to. .	Mitigation includes: • Flexible disposal strategy in place to handle uncertainty in UG volumes of water. • Grouting of critical abandoned boreholes planned for completion in Q4 2023 • Mine dewatering capacity for storm water. • Updated 3D Hydrogeological Model nearing completion. • Conduct hydraulic investigation and monitoring as soon as UG access becomes available • Update the groundwater flow model and inflow prediction immediately after these data become available • Continue to conduct probe drilling to identify any upcoming ground water issues and/or grouting needs
Flooding UG mine, specifically submerging the main UG pump station on 285 mL	Water Risks	If a combination of 200-year storm event and/or unexpected groundwater inflow beyond the 3-D Hydrogeological model takes place, the UG water pumping capacity may be insufficient and lead to flooding of the mine. Flooding could occur as a result of: • Extended power failure renders dewatering pumps inoperable • Mechanical failure of dewatering infrastructure-ruptured shafts dewatering pipes • More GW inflows than initially planned • UG Mine pumping capacity and/or flood chamber capacity insufficient. • Blasting of crown pillar allowing stormwater from the OP reporting to the UG. Submerging pressure pumps at Level 285 could stop mining for a long period; removing water from inundated areas could be a significant task. There may also be major damages to crusher and conveyors; loss of revenues and major capital cost to reactivate the mine.	There are several mitigation efforts underway or in place including: • Design of 26,000 m^3 of flood capacity below the extraction level • Basic engineering completed for main UG pump station @ 285 L. • Diesel generator back-up power in case of grid failure • Early establishment of pumping systems Levels 670 L and 470 L • Establishment of sumps and pumps in front of water intersections • Water-tight doors to be designed and installed to garner additional flood capacity UG • Lateral Development Grouting Plan being developed • Option in place to pump from 470 (or 310) straight to surface. • Revised dewatering capacity for the main UG Pump station @285 L • UG refuge chamber accessible by workers under all conditions. • Water in-flow mitigation/grouting abandoned boreholes.
Safety Risks-Significant fall of ground during construction	UG Development Risk	If a significant fall of ground occurs during construction, especially if it leads to equipment damage or injuries to workers, there could be an extended delay to construction due to time needed for recovery, investigations, ground support and re-start.	Ground control problems are being mitigated by: • Daily visits by a geotechnical engineer and pre-shift inspections and scaling by crews and persons-in-charge • An established ground control management plan is in place and is being followed and audited regularly for compliance

26.3 Opportunities

Several opportunities have been identified during the FS that could improve project economics, reduce risk or improve execution. Table 26-2 highlights some of the more significant opportunities that will be or are being explored.

Table 26-2: Identified Project Opportunities

Opportunity	Explanation
Optimization of the final OP benches	Preliminary investigations are underway to review options for steepening the lower OP slopes to allow for increased ore tonnes to be mined. Wall stabilization measures would likely be required if this option is pursued but the extra cost may be worthwhile to defer stockpiled ore processing.
Reduced shaft cost and duration	Several cost saving initiatives are currently underway to decrease the construction duration of the shaft, save material costs, defer non-critical capital expenses and lower the overall cost of the shafts. Over the past year, several shaft sinking optimizations were implemented including but not limited to the following examples: • Lengthening drill Jumbo slides to drill longer rounds • Using mixer trucks to increase the speed and quality of concrete liner placement • Modifications to grouting operations • Improved kibble design and increased kibble capacity
Mining below 310 masl down to 250 masl and below	Approximately 1.8 Mt of ore, mainly high-value EM/PK(S) is below the currently planned mine between 250 masl and 310 masl. This portion of the indicated resource has not been included in the UG FS. This 60 m vertical could add high-value material from a sub-level caving method after the currently planned production is complete. There are over 300,000 ct in this zone. The inferred resources is modelled down to 60 masl and could be drilled and potentially upgraded to the Indicated classification. The resource is open at depth, but the pipe reduces in diameter with depth.
Increased production rate after stope blasting is complete in	Once drilling and blasting is complete, production from UG can be increased to >3.1 Mt/a as only mucking, crushing and hoisting will be required.
Recovery of exceptional diamonds	KDM's robust and proven diamond size-distribution model predicts that exceptional diamonds will continue to be recovered at a predictable rate. Their contribution to the diamond price model has not been taken into account in the FS, adding significant upside potential on the mine revenue stream.

Opportunity	Explanation
Lower Groundwater Inflow Rate	If the Mea and granite are less permeable than what are simulated in the model, lower inflow could be expected. Reduced CAPEX and OPEX of pumping water from UG, water treatment, and disposal. The number of UG drainage holes could also be reduced.
Tailings Facilities	The TSFs have been built with extra storage capacity to minimize the rate of rise; to allow deposition flexibility and adaptability; improved consolidation times; and minimize the chances of the dam overtopping. The facility can support increased mine life.

27 RECOMMENDATIONS

The KDM UGP is economically viable and should continue with completing detailed engineering, UG mine and infrastructure construction and transition to UG production. The cost of the advancement of the UGP is detailed in the CAPEX section. Specific technical recommendations are summarized below and within the current scope of the Project.

27.1 Geotechnical

The study has addressed the critical risks, but at this stage of the study some uncertainties remain. Additional predictive modelling is suggested to improve confidence in the design, such as incorporating evolving pore pressures from the hydrogeological model at a higher resolution, to obtain a more accurate estimate of potential overbreak and sloughing, and to remove the kimberlite skin during the mining sequence.

Monitoring of the performance of the stoping will be essential to verify the rock mass characteristics and to address any unexpected behaviour:

- Stress measurements using the overcoring method during development;
- Measuring the length of blastholes after each blast and inspecting selected blastholes with borehole cameras;
- Lidar cavity monitoring through drillholes at intervals to evaluate potential overbreak and sloughing; and
- Monitoring overbreak and sloughing from rim drives and access drives where stoping has been completed.

Adoption of good draw management is essential to minimize dilution and the risk of mud rushes and air blasts.

FLAC3D models should be updated after completing the stress measurements and during the life of mine and calibrated against actual rock mass performance.

27.2 Hydrogeology

Based on the available data and potential uncertainty of the model calibration and predictions, the following recommendations are made:

- Assess whether the existing and planned dewatering/depressurization infrastructure will continue to achieve the slope stability requirements over the life of both the OP and UG mining operation;

- Install groundwater level monitoring points as soon as UG access is available. These monitoring points are designated sub-vertical drillholes that are equipped with collar casing and pressure valves;

- In addition to the monitoring points, the Mine should also plan to install pressure valves in selected cover holes or drain holes to increase the monitoring points of the groundwater levels in the Mea and granite units;

- During the development of the UG workings, measure the groundwater flow rate in the UG and Maintain an accurate water balance model of the UG mine;

- The implementation of the UG infrastructure should consider the effect of high salinity; and

- The groundwater flow model should be updated on a yearly base to update the predicted inflow rate.

27.3 Geology and Mineral Resources

Recommendations for further work to increase confidence in key areas and continue to advance the understanding of the Mineral Resource include the following:

- Mapping of development drives and geological assessment to determine the distribution of the KIMB3 unit and the variant of the of EM/PK(S) intersected in UGP drilling below 500 masl;

- Mapping of development drives in kimberlite to better constrain the modelled but not drill confirmed extent of the M/PK(S) domain below 438 masl elevation;

- Continued incorporation of pit geological mapping to enhance pipe contact and internal kimberlite domain definition;

- Monitoring and refinement of the SFD for the M/PK(S) and EM/PK(S) as development ore and production ore is introduced into the plant, and where and when possible, based on discrete production data; and

- Continued reconciliation of production forecasts relative to mine production.

27.4 Environmental and Permitting:

- Complete an Environmental Impact Statement and confirm regulatory approval for the proposed on-site storage and mechanical evaporation of produced saline groundwater in a lined pond between 2026-2030, and develop disposal plans from 2030 onwards;

- KDM should conduct a pre-construction archeological survey and include archeological monitoring during construction of TMF/Slime Dam 3. Similarly, KDM should conduct a biodiversity survey focused on identifying and, if present, transplanting the Devils claw

(Harpagophytum procumbens) and the Hoodia (Hoodia currorii), considered threatened according to Bostwana's "Red Book";

- Given requirements related to the Equator Principles, a growing focus on climate change risks, and Botswana's plan to reduce its national carbon footprint, Lucara should continue to explore opportunities to reduce its GHG emissions and implement its Climate Action Plan;
- Lucara should safekeep soil stockpiles, record soil volumes stockpiled, explore use of sand to supplement topsoil, and other measures; and
- Good international industry practice requires incorporation of socio-economic considerations (for mine reclamation and closure planning). It is recommended that Lucara incorporates socio-economic considerations in its future closure planning processes.

27.5 Tailings

Design, construction, and operation of tailings storage facilities (TSFs) requires a holistic approach that integrates best practices, innovative technologies, and continuous improvement processes. Continue with adoption and implementation of GISTM.

- Engineering Design:
  - Engagement of specialist consultants for geotechnical and hydrological modelling techniques to optimize TSF design for stability, containment, and environmental protection; and
  - Utilize advanced dewatering technologies, such as thickening and filtration, to reduce water content in tailings and improve storage efficiency.
- Operational Monitoring:
  - Use of comprehensive monitoring systems, including instrumentation for geotechnical stability, water quality, and environmental parameters, to detect early signs of potential issues; and
  - Implementation of real-time monitoring and data analytics platforms to enable proactive decision-making and rapid response to operational challenges or anomalies.
- Sustainable Tailings Management Practices:
  - Adoption of sustainable tailings management practices; and
  - Explore opportunities for tailings reprocessing or utilization.

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29 UNITS OF MEASURE, ABBREVIATIONS AND ACRONYMS

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
Acfm	actual cubic feet per minute
ACK	apparent coherent kimberlite
ADT	articulated dump truck
AG	autogenous
ALT	active layer thickness
ALT	active layer thickness
AMD	acid mine drainage
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
AWR	all-weather road
B	billion
BD	bulk density
BPC	Botswana Power Corporation
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
BWP	Botswana Pula (Botswana currency)
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
CAPEX	Capital Expenditure
CCS	consequence classification
cfm	cubic feet per minute
CHP	combined heat and power plant

Symbol / Abbreviation	Description
CIM	Canadian institute of Mining and Metallurgy
CK	coherent kimberlite
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cP	centipoise
c/s	carats per stone
c/t	carat per tonne
Cr	chromium
CRD	coarse residue deposition
ct or cts	carat
Cu	copper
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DGPS	differential global positioning system
diam	diameter
DMS	dense media separation
dmt	dry metric ton
DRA	Dowding, Reynard and associates
DTC	diamond trading company
DWT	dead weight tonnes
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
EM/PK (S)	eastern magmatic pyroclastic kimberlite
EOR	engineer of record
EPCM	engineering, procurement, and construction management
ERD	explosives regulatory division
FEL	front-end loader
FRD	fine residue deposit
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second

Symbol / Abbreviation	Description
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GHG	greenhouse gas
GISTM	Global Industry Standard on Tailings Management
GJ	gigajoule
GPa	gigapascal
gpm	gallons per minute (us)
GTZ	glacial terrain zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m2)
ha	hectare
HG	high grade
HK	hypabyssal kimberlite
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
hrs	hours
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
ICMM	International Council on Mining and Metals
in	inch
in^2	square inch
in^3	cubic inch
IR	infrared
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity

Symbol / Abbreviation	Description
k	kilo (thousand)
KDM	Karowe Diamond Mine
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
KIM	kimberlitic indicator mineral
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
KP	Knight Piésold
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LDR	large diamond recovery
LG	low grade
LGM	last glacial maximum
LOM	life of mine
m	metre
M	million
m/day	metres per day
m/min	metres per minute
M/PK(S)	magmatic pyroclastic kimberlite
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/day	cubic metres per day
m^3/h	cubic metres per hour

Symbol / Abbreviation	Description
m^3/s	cubic metres per second
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
masl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
MBP	melt-bearing pyroclasts
mbs	metres below surface
mbsl	metres below sea level
MCA	multi criteria analysis
Mct	million carats
MDR	mega diamond recovery
mg	milligram
mg/L	milligrams per litre
MIDA	microdiamond
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMSIM	metamorphosed massive sulphide indicator minerals
mo	month
MPa	megapascal
MSC	Mineral Services Canada Inc.
Mt	million metric tonnes
Mt/a	million metric tonnes per annum
MVA	megavolt-ampere
MW	megawatt
MWh	megawatt hour
Nc	critical speed
NG	normal grade
NGL	natural ground level

Symbol / Abbreviation	Description
Ni	nickel
NI 43-101	National Instrument 43-101
Nm^3/h	normal cubic metres per hour
NPV	net present value
NQ	drill core diameter of 47.6 mm
OP	open pit
OPEX	operational expenditure
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PAR	population at risk
PDC	process design criteria
PEA	preliminary economic assessment
PFK	processed fine kimberlite
PFS	preliminary feasibility study
PGE	platinum group elements
PK	pyroclastic kimberlite
PLL	potential loss of life
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
PSD	particle size distribution
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
R/O	reverse osmosis
RC	reverse circulation
RH	Royal Haskoning
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
RVK	resedimented volcaniclastic kimberlite
s	second (time)
Scfm	standard cubic feet per minute

Symbol / Abbreviation	Description
SABS	South African Bureau of Standards
SANS	South African National Standards
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SFD	size frequency distribution
SG	specific gravity
SQ	square
SRC	Saskatchewan Research Council
SRK	SRK Consulting Inc.
st/kg	stones per kilogram
st/t	stones per metric tonne
SWD	stormwater dam
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
tCO_{2e}	tonnes of carbon dioxide equivalent
TCR	total core recovery
TDBA	tailings dam breach assessment
TFFE	target for further exploration
TMF	tailings management facility
TSF	tailings storage facility
t/h	tonnes per hour
t/m	tonnes per metre
t/m^3	tonnes per cubic metre
ts/hm^3	tonnes seconds per hour metre cubed
US	United States
US$	dollar (American)
UTM	universal transverse mercator
V	volt
v/v	volume/volume
VEC	valued ecosystem components
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight

Symbol / Abbreviation	Description
wk	week
wmt	wet metric ton
WRSF	waste rock storage facility
WRSF	waste rock storage facility
wt	weight
XRT	x-ray transmission
ø	diameter
µm	microns
µm	micrometre

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000

Rock Type / Unit	Description
PK	Pyroclastic Kimberlite
RVK	Resedimented Volcaniclastic Kimberlite
VK	Volcaniclastic Kimberlite
ACK	Apparent Coherent Kimberlite
BBX	Country rock breccia
CBBX	Calcretized country rock breccia
CFK(C)	Carbonate-rich fragmental kimberlite (Centre Lobe)
CK	Coherent Kimberlite
CKIMB	Calcretized kimberlite
CRX	Country rock xenolith
EM/PK(S)	Eastern magmatic/pyroclastic kimberlite (South Lobe - main pipe infill)
FK(N)	Fragmental kimberlite (North Lobe)

Rock Type / Unit	Description
HK	Hypabyssal Kimberlite
INTSWBAS	Large internal block of basalt
KBBX	Kimberlite and country rock breccia
KIMB1	Volumetrically minor hypabyssal kimberlite
KIMB3	Minor hypabyssal kimberlite; increasing volume below 500 masl
KIMB4a	Localized variant of EM/PK(S)
KIMB5	Volumetrically minor hypabyssal kimberlite
KIMB6	Volumetrically minor hypabyssal kimberlite
KIMB7	Volumetrically minor kimberlite
LSTX	Paleozoic carbonate xenolith
M/PK(S)	Magmatic/pyroclastic kimberlite (South Lobe - main pipe infill)
WBBX	Weathered country rock breccia
WM/PK(S)	Western magmatic/pyroclastic kimberlite
WK	Weathered kimberlite