LUCARA DIAMOND CORP.
KAROWE DIAMOND MINE
2025 FEASIBILITY STUDY TECHNICAL REPORT

LOCATION: BOTSWANA
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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.

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1 EXECUTIVE SUMMARY

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Lucara Diamond Corp. (Lucara or the Company) to carry out an updated Feasibility Study (FS) for the Karowe Diamond Mine (Karowe or KDM) Underground Mine Project (UGP or Project) currently being built to establish underground (UG) mining after the completion of the current open pit (OP) mining. This 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 $) and all elevations refer to metres above sea level (masl) unless otherwise noted.

This Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 and Form 43-101F1 (collectively referred to as NI 43-101), and was prepared using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for mineral resources and mineral reserves (CIM Definition Standards, 2014), and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM Best Practice Guidelines, 2019).

This Report is updated from the 2023 FS Technical Report (2023 FS) published on March 12, 2024 (Doerksen et al., 2024) and encompasses the following modifications:

- Project construction progress (surface infrastructure and underground development) to September 30, 2025, and in particular the achievement of successfully reaching both production and ventilation shaft bottoms and lateral development connecting the two shafts;
- Revised economic model with updated exchange rates and diamond prices, inclusion of Legacy stone values, exclusion of $436 million in costs incurred to date and update of financing costs;
- Re-baselining the UGP schedule and, as a result, the open pit mine and processing facility production plans;
- Re-estimation of operations budgets and project capital and operating cost projections;
- Geological model updates incorporating pit mapping and drillhole data from five core holes (1,781 m) drilled between July and September 2025;
- Updated Mineral Resource Estimate (MRE);
- Hydrogeological and geomechanical model updates;
- Mine design, method, and schedule updates;
- Advancement of detailed engineering designs; and
- Groundwater management on surface.

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

- **SRK Consulting (Canada) Inc.**: Geology and Mineral Resource estimation;
- **Middindi Consulting**: Geotechnical analysis,
- **Beck Engineering:** Geomechanical Modelling and Cave Simulations;
- **DRA Botswana (Pty) Ltd.:** Mineral processing;
- **K-Met Consultants Inc.**: Metallurgical testing;
- **Itasca Denver**: Hydrogeological modelling;
- **Knight Piésold (Pty) Ltd. (Botswana) (KP):** Waste material management; and
- **PRIZMA LLC**: Environment and social considerations.

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 2.85 million tonnes per annum (Mt/a) of feed to the processing plant. KDM has processed over 40 million tonnes (Mt) of ore and sold over 4.9 million carats (Mct) of diamonds since the start of operations.

The mine has established itself as one of the world's most prolific producers of large, gem quality, Type IIa diamonds. Karowe is the only diamond mine to have recovered nine diamonds in excess of 1,000 carats (ct) in weight. The mining of kimberlite from the KDM has produced forty-nine (49) large, gem-quality diamonds that were recovered and sold in the period March 2013 to November 2024, for $5 million or more each. These high-value gemstones are designated as Legacy diamonds by Lucara and the 49 recovered and sold over the 12-year period account for $557.3 million in total revenue. Roughly 67% of the mine's revenue is generated by +10.8 ct diamonds (Specials) that make up greater than 7% of the carats produced (Lucara, 2025c).

The in situ OP reserve is planned to be fully depleted by mid-2026. The mine currently has approximately 7.6 Mt of stockpiled reserves. This FS evaluates extending the project life by establishing UG mining production after depletion of the OP. The stockpiled reserves 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 UGP is summarized as follows:

- UG Mining:
  - Blind sinking an 8.5 metre (m) finished diameter Production Shaft (P/S) approximately 740 m deep equipped to hoist a nominal 7,800 tonnes per day (t/d) of ore and additional development waste;
  - Blind sinking a 6.0 m finished diameter unequipped Ventilation Shaft (V/S);
  - Bottom-up bulk stoping utilizing mass long hole shrinkage (LHS) mining – a form of blast assisted caving (BAC) - until geomechanical conditions allow for free caving through to the bottom of the OP;
  - Hoisting of 28.5 Mt of UG ore mined at a grade of approximately 12.9 carats per hundred tonnes (cpht) providing 3.7 Mct recovered; and
  - Extraction of approximately 355 m vertical of the South Lobe, of the AK6 kimberlite from 310 masl (700 m below surface (mbg)) to the bottom of the depleted OP (approximately 665 masl or 350 mbg).
- Processing ore through the existing processing plant at a throughput of 2.85 Mt/a;
- An eight-year UG construction period that commenced in 2020 and ends in 2028; and
- 10 years of planned UG operations from 2028 through 2038.

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 or by UTM coordinates: 341,590 m East and 7,621,640 m South.

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 town 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 Licences 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 Licence 2008/6L.

1.4 History, Exploration and Drilling

The AK6 kimberlite was discovered by De Beers in 1969. Since its discovery, there have been a multitude of exploration and resource definition programs completed on the property by De Beers, the Boteti JV and Lucara. The most significant programs are outlined in Table 1-1. No exploration other than core drilling and sampling has been carried out on the AK6 kimberlite by Lucara since acquiring the project in 2009. The current resource estimate is based on data collected during these programs, incorporating results from mining operations and diamond sales since the commencement of commercial production in 2012. Since the 2023 FS was completed, an additional five core holes have been drilled. This drilling was primarily for geotechnical and geological characterisation of KIMB3 and the six additional minor units which constitute newly estimated minor-volume domains in the September 2025 MRE, and to inform placement of UG infrastructure.

Table 1-1: Historical Exploration and Drilling Programs by Previous Owners and Lucara

Owner	Stage	Work Completed	Duration
De Beers and Boteti JV	Early Evaluation	5 x 12¼" large diameter drillholes totalling 679 m, 97 t bulk sample	2003 - 2005
		DMS and diamond recovery	
		Geophysical surveys	
Boteti JV	Phase 1 Advanced Exploration	44 x 6½" percussion holes for delineation totalling 4,575 m	2005 - 2006
		12 x vertical core holes (NQ) as LDD pilots, totalling 2,980 m	
		17 x inclined core holes (NQ) for delineation totalling 6,901 m	
		13 x 23" LDD totalling 3,699 m	
		DMS processing and diamond recovery from 1,775 t	
Boteti JV	Phase 2 Advanced Exploration	11 x vertical core holes (NQ) as LDD pilots totalling 4,184 m	2006 - 2008
		29 x inclined core holes (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	
Lucara	Delineation & Geotechnical Drilling	15 x core holes (HQ and NQ) totalling 12,272 m	2016 - 2017
		916 microdiamond samples (7,315 kg)	
Lucara	Delineation & Geotechnical Drilling	37 x core holes (HQ and NQ) totalling 23,958 m	2018 – 2019
		153 microdiamond samples (1,232.8 kg)	
Lucara	Shaft Geotech Investigation	2 x core holes (NQ) totalling 1,514 m	2020

Owner	Stage	Work Completed	Duration
Lucara	Delineation & Geotechnical Drilling	5 x core holes (HQ and PQ) totalling 1,781 m	2025

Since the onset of commercial production to the end of Q3 2025, the Karowe Mine has produced 5.08 million carats (Mct) from 34.8 Mt of processed kimberlite and has sold via tender a total of 4.87 Mct for a total revenue of $2.65 billion, resulting in an achieved sold average price of $545/ct.

1.5 Geology and Mineralization

KDM is exploiting the AK6 kimberlite which is part of the mid-Cretaceous Orapa Kimberlite Field (OKF) in the Central District of Botswana on the western edge of the Zimbabwe Craton. The OKF comprises 86 known kimberlite bodies of post-Karoo age. Nine of the kimberlites have been or are currently being mined – AK1 (Orapa), AK6 (Karowe), BK1, BK9, BK12, BK15 (Damtshaa), DK1 and DK2 (Letlhakane), and BK11.

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 (~125 m thick), underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup (~350 m thick), below which is the granitic basement.

The KDM structural geology model was updated in 2024 to provide a robust foundation for pit steepening initiatives, risk assessments, and underground design at Karowe by identifying key structures and structural domains influencing pit stability and underground development. The model incorporates previous pit mapping and drillhole structural data, pit wall scans, drone imagery, and new pit and underground mapping data collected during 2024.

The structural domain encompassing the central axis of the kimberlite pipes is characterized by a zone of pervasive NNW trending continuous faults and joints known as the ‘fracture corridor’. The dominant steeply dipping NW to NNW strike slip faults are associated with kimberlite dykes aligned with the NNW structural framework. Steeply dipping EW to NE trending faults concentrated near the kimberlite pipes are interpreted as post-emplacement features. Low-angle discontinuities within the South Lobe are interpreted as relaxation features along pre-existing lava flow contacts

AK6 comprises three adjacent, geologically distinct, coalesced kimberlite pipes known as the South, Centre and North Lobes. The pipes trend roughly north-south with a combined undepleted surface expression of ~3.3 ha and maximum area of ~8 ha at 120 m below surface. 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 constitutes the current in situ resource. The pipe contacts are sharp and distinct, and there is minimal or no brecciation and minimal alteration of the country rock at the contacts.

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 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. The calcretized and weathered horizons in the upper portions of the lobes have been mined out. The South Lobe comprises two volumetrically dominant units, Magmatic/Pyroclastic Kimberlite (M/PK(S)) and Eastern Magmatic /Pyroclastic Kimberlite (EM/PK(S)), and several 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.

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) units are fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly homogeneous to locally crudely stratified macrocrystic coherent or apparent coherent kimberlites. They are distinguished based on differences in olivine, primary groundmass, country rock xenolith and mantle xenocryst populations. Both units exhibit textural, component and large-scale characteristics suggesting they formed extrusively. The presence of low-angle discontinuities between thick vertical packages of homogeneous coherent kimberlite, irregular columnar jointing, pillow structures, and tabular/blocky fractured zones display similarities to features observed in some basaltic lava flows.

The regular morphology of the South Lobe, the presence of coherent kimberlite over a significant vertical extent (900 m drill confirmed) and the characteristics of the two main pipe-filling coherent kimberlite units, M/PK(S) and EM/PK(S), indicates that they formed by emplacement of lava flows and fountains into the previously excavated pipe to form lava lake deposits. The M/PK(S) and EM/PK(S) were later intruded by various small-volume hypabyssal kimberlites (e.g. minor units KIMB1, KIMB3), mainly along internal and pipe contacts.

The Centre and North Lobes both comprise volcaniclastic (historically ‘fragmental’) to apparent coherent (historically ‘magmatic’) kimberlite. Mining has ceased in both lobes, with the remaining in situ kimberlite being excluded from the updated MRE as Unclassified material.

The geological model of AK6 consists of a pipe shell model defining the geometry and extent of the deposit and an internal geological domain model comprising multiple wireframes representing the spatial distribution of the various units. The current geological model (Figure 1-1) was first presented in Doerksen et al. (SRK, 2019; Doerksen et al., 2019) as an update to the Nowicki et al. (2018) model based on the 2018/2019 FS drilling program. The pipe shell model has been updated in September 2025 for mining exposures of the contacts (all lobes) and at depth in the South Lobe using new pierce points from the 2025 core drilling program. Previously, the minor kimberlite units other than KIMB3 were not resolved as discrete domains in the model due to their small volumes and generally discontinuous and dispersed distributions and hence were left as formally undefined minor volumes within the main domains. In the current updated model, the other minor units have been modelled separately to enable their volumes to be estimated and to improve knowledge of their geotechnical parameters. Between the end-September 2025 pit to 250 masl, the M/PK(S) and EM/PK(S) combined make up 94% of the pipe volume, and the minor units together make up 6% of the pipe volume.

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.

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

Lobe	Unit	Domain	Description	Status at September 2025
South	BBX	BBX(S)	Country rock breccia	Mined out
	CBBX	CBBX(S)	Calcretized country rock breccia	Mined out
	CKIMB	CKIMB(S)	Calcretized kimberlite	Mined out
	INTSWBAS	INTSWBAS(S)	Large internal block of basalt	Mined out
	WBBX	WBBX(S)	Weathered country rock breccia	Mined out
	WK	WK(S)	Weathered kimberlite	Mined out
	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite	
	EM/PK(S)	M/PK(S)	Eastern magmatic/pyroclastic kimberlite	
	KIMB3	KIMB3	Minor hypabyssal kimberlite; volume increases below 500 masl	
	WM/PK(S)*	WM/PK(S)	Western magmatic/pyroclastic kimberlite	
	KIMB1*	KIMB1	Volumetrically minor hypabyssal kimberlite	
	KIMB4a*	KIMB4a	Localized variant of EM/PK(S)	
	KIMB5*	KIMB5	Volumetrically minor hypabyssal kimberlite	
	KIMB6*	KIMB6	Volumetrically minor hypabyssal kimberlite	
	RFW-ALT*	RFW-ALT	Highly altered kimberlite of unknown type	
	KIMB7	n/a	Volumetrically minor kimberlite	
Centre	BBX	BBX(C)	Country rock breccia	Mined out
	CKIMB	CKIMB(C)	Calcretized kimberlite	Mined out
	KBBX	KBBX(C)	Kimberlite and country rock breccia	Mined out
	WBBX	WBBX(C)	Weathered country rock breccia	Mined out
	WK	WK(C)	Weathered kimberlite	Mined out
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite	Mined out
	FK(C)	FK(C)	Fragmental kimberlite	
North	BBX	BBX(N)	Country rock breccia	Partly mined out
	CKIMB	CKIMB(N)	Calcretized kimberlite	Mined out
	KBBX	KBBX(N)	Kimberlite and country rock breccia	Mined out
	WBBX	WBBX(N)	Weathered country rock breccia	Mined out
	WK	WK(N)	Weathered kimberlite	Mined out
	FK(N)	FK(N)	Fragmental kimberlite	

Notes:

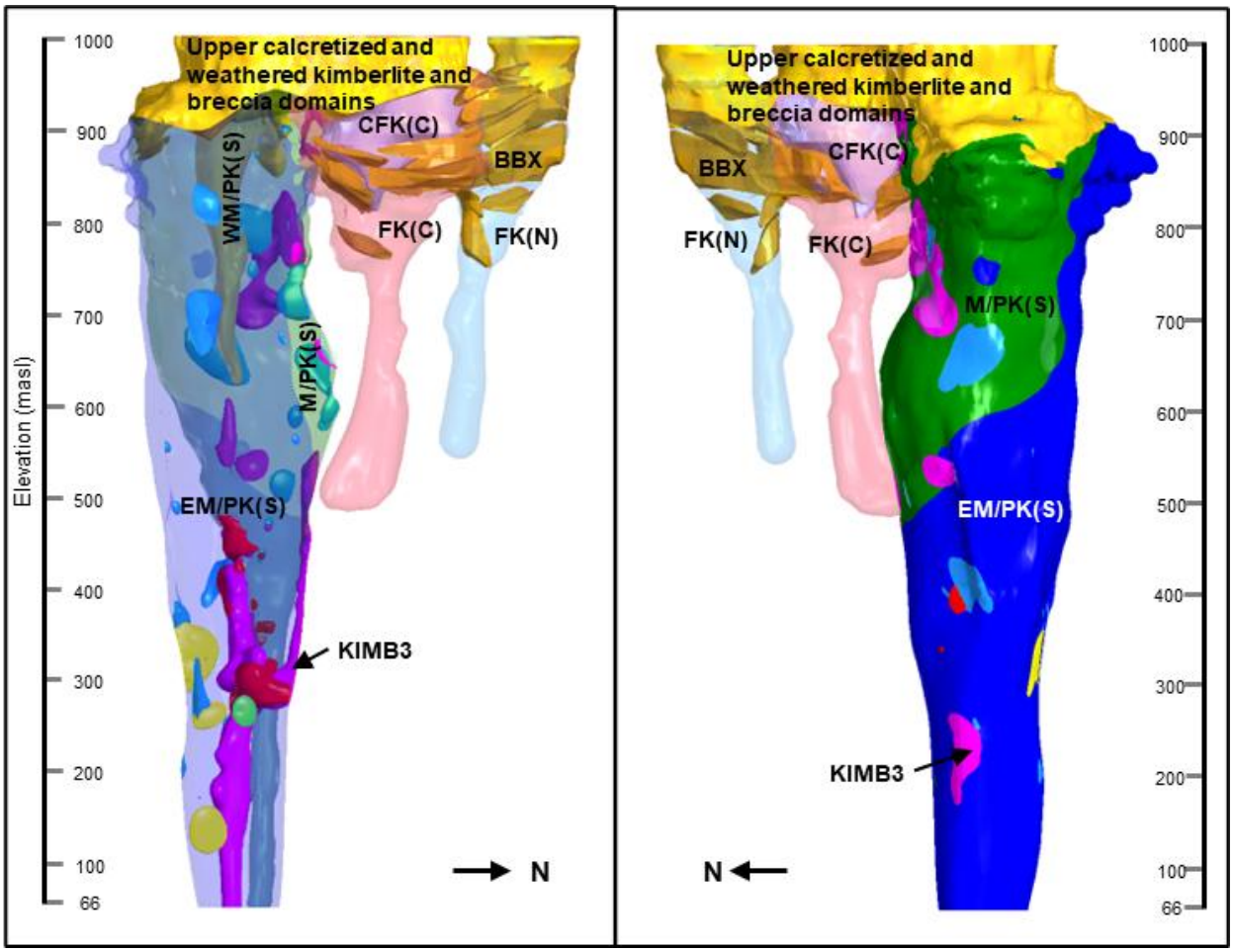
*These minor units were previously included in the major domain models and have now been modelled as separate domains.

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

The historical term ‘breccia’ meaning country rock xenolith-rich has been maintained for continuity with previous reporting.

Source: SRK (2025)

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



Source: SRK (2025)

KDM is one of the world’s most significant producers of large and high-value diamonds including fine-gem Type IIa and rare coloured diamonds. From inception to the end of Q3 2025, Karowe has recovered 43 diamonds > 300 ct, 69 diamonds between 200 and 300 ct, and 283 diamonds between 100 and 200 ct. The mine has recovered nine diamonds in excess of 1,000 ct since

2015. The South Lobe has consistently recovered high-value diamonds in excess of 200 ct and several of the Legacy stones have come from the EM/PK(S), including the 1,109 ct Lesedi La Rona.

The M/PK(S) and EM/PK(S) each have distinct diamond grades, size frequency distributions (SFD), average $/ct diamond values and significant stone recoveries. The SFDs are considered as coarse with roughly 60 to 70% of the mine's revenue being generated by +10.8 ct diamonds (Specials) that make up greater than 6% of the carats produced.

Diamonds at Karowe are classified into four primary mantle source suites: eclogitic, peridotitic, sublithospheric and websteritic (Motsamai et al., 2018). The occurrence of superdeep mineral inclusions in Karowe diamonds is consistent with a sublithospheric origin for the exceptionally large diamonds.

1.6 Mineral Processing Testwork

An assessment of the plant capacity when treating UG ore was conducted by testing X-ray transmission sorting and ore hardness of the deeper UG ore compared to the OP 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 no significant difference in the hardness between EM/PK(S) and M/PK(S). The samples demonstrated similar characteristics to the OP material processed in the existing AG mill, and therefore, the current comminution circuit at the same production rate should be able to process the UG material planned to be mined.

1.6.2 XRT Testwork

The predominant diamond separation and extraction process in the existing 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 analyze the atomic density of materials and then physically separate the materials with a diamond / carbon signature from non-diamondiferous material.

The UG is planned to mine kimberlite through a carbonaceous shale host lithology. It is expected that some carbonaceous shale will report as dilution to the mill and potentially the XRT bulk sorters 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 Sorting Solutions laboratory in Germany.

Results of the First Inspection Report tests completed by Tomra determined that the coal and carbonaceous shales using a standard sorting algorithm for 16-32 mm feed material, as well as all other host waste rock lithologies tested could potentially be identified and separated by the XRT machines from the diamonds. Due to the shales and coal being broken down to smaller than 4 mm in the comminution circuit it is unlikely they would report to the downstream XRT's and impact recovery. To successfully separate the diamonds, additional sampling and testwork using the proposed UG ore is recommended to ensure the current Tomra system is programed to correctly recognize the waste from the diamonds.

1.7 Mineral Resource Estimate

The September 2025 Mineral Resource Estimate (MRE) for KDM utilizes back-captured and recompiled daily open pit production data for the period January 2016 to September 2025 to reconcile mineral resource metrics estimated in June 2023 with actual diamond grades, size frequency distributions and sales prices obtained for diamonds recovered from the dominant M/PK(S) and EM/PK(S) domains in the South Lobe. The reconciliation ultimately considers 4,700 kilotonnes (kt) of pure M/PK(S) and 553 kt of pure EM/PK(S) kimberlite processed during January 2020 to September 2025 to validate and endorse the June 2023 global grade estimates of, respectively, 10.8 and 21.0 cpht, though subject to a 1.9% grade under-recovery that likely represents diamond losses to process plant tailings. Continuity of these global grade estimates for the M/PK(S) and EM/PK(S) domains to a depth of 250 masl (760 mbg) was established in prior work (Doerksen et al., 2024), none of which is materially affected by limited drilling completed during Q3 2025. The latter drilling was directed primarily at geotechnical and geological characterization of the KIMB3 and six additional minor geological units (KIMB1, KIMB4a, KIMB5, KIMB6, WM/PK(S) and RFW-ALT); the latter six constitute newly estimated minor-volume domains in the September 2025 MRE.

Depletion of the KDM open pit to approximately 715 masl (as of end-September 2025) has left North and Centre Lobe kimberlite stranded in locations that Lucara does not currently deem economically accessible by open pit or underground mining methods. As a result, the remaining North and Centre Lobe kimberlite volumes have been relegated to an Unclassified status and do not appear in the current MRE. The most recent consequential production from the North Lobe occurred during February 2023, and during February 2025 for the Centre Lobe.

The daily open pit production data back-captured and recompiled for the period January 2016 to September 2025 enabled tracing of diamonds recovered from South Lobe M/PK(S) and EM/PK(S) production through parcel consolidation to final sales revenue generated via the three sales channels utilized by Lucara: HB Antwerp, diamond tenders and the Clara platform. Reviews

of the sales data during February 2025 and October 2025 provide updated actual average US$ per carat prices as shown in Table 1-3.

The 2025 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 September 30, 2025.

Table 1-3: KDM 2025 Mineral Resource Statement (effective date of September 30, 2025)

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mct)	Grade (cpht)	Average (US$/ct)
Indicated	South_M/PK(S)	5.28	15.70	2.97	1.70	10.8	520
	South_EM/PK(S)	5.97	17.44	2.92	3.66	21.0	695
Total Indicated		**11.25**	**33.13**	**2.95**	**5.36**	**16.2**	**639**
Inferred	South_M/PK(S)	0.10	0.30	2.97	0.03	10.5	520
	South_EM/PK(S)	1.39	4.16	3.00	0.87	20.9	695
	South_KIMB3	0.45	1.20	2.68	0.13	10.9	
	South_WM/PK(S)	0.13	0.33	2.62	0.02	5.0	
	South_KIMB4a	0.10	0.30	2.95	0.05	15.0	
	South_KIMB1	0.07	0.20	2.74	0.03	15.0	
	South_KIMB5	0.06	0.17	2.96	0.02	10.0	
	South_KIMB6	0.00	0.01	2.96	0.00	15.0	
	South_RFW-ALT	0.02	0.04	2.35	0.00	10.0	
Total Inferred		**2.32**	**6.72**	**2.9**	**1.15**	**17.2**	

Notes:

1. Prepared by Hermanus Grütter, P.Geo, Ph.D. of SRK Consulting (Canada) Inc.;
2. CIM definitions were followed for Mineral Resources;
3. Effective date of September 30, 2025;
4. All numbers have been rounded to reflect accuracy of the estimate;
5. Mineral Resources are stated above a +1.25 mm bottom cut-off;
6. Average diamond value estimates are based on diamond sales data provided by Lucara Diamond Corp.;
7. Mineral Resources are in situ and inclusive of in situ Mineral Reserves;
8. Mineral Resources are exclusive of all mine stockpile material;
9. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery;
10. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability; and
11. 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.

Source: SRK (2025)

Based on past resource estimation practices, the average prices shown in Table 1-3 exclude revenue generated from infrequently recovered large, fine-gem diamonds that Lucara designates as Legacy diamonds, and which sell for US$5 million or more each. Forty-nine such Legacy diamonds have been recovered and sold in the period March 2013 to November 2024 and form the basis of a proprietary Revenue Frequency Distribution (RFD) model for Legacy diamonds that was developed and reported on in October 2025 (SRK 2025). The RFD model imposes a cap of US$25.62 million per Legacy diamond, reproduces (capped) aggregate 2013 to 2024 revenue of US$469.4 million for South Lobe M/PK(S) and EM/PK(S) remarkably well, and attributes (capped) mean annual Legacy diamond revenue of US$43.0 million per million tonnes (Mt) processed for the EM/PK(S) and US$13.7 million per Mt processed for the M/PK(S) (Table 1-4).

Table 1-4: Summary of 2025 Revenue Frequency Distribution Model for >US$5 million Legacy Diamonds

Item	Period	M/PK(S)	EM/PK(S)	Centre Lobe	Total
Legacy Diamond Count (n =)	2013-2024	29	17	3	49
Actual Revenue (US$M)	2013-2024	255.4	284.9	17.0	557.3
Capped Revenue (Capped US$M)	2013-2024	255.4	210.6	17.0	483.0
RFD Model Revenue (Capped US$M)	2013-2024	261.9	207.5	-	
Revenue Variance (RFD Model / Capped Revenue, %)	2013-2024	+2.5%	-1.5%	-	
Mean Attributable RFD Model Revenue/ Mt processed (Capped US$M / Mt)	2015-2024	13.7	43.0	-	

Notes:

Mean Attributable RFD Model Revenue / Mt processed can be applied when reasonable prospects exist of processing more than 2.0 Mt of combined M/PK(S) and EM/PK(S). The potential exists of underperforming (or overperforming) Mean Attributable RFD Model Revenue at lower tonnage processed, to the extent that zero Legacy diamonds could be recovered.

Source: SRK (2025)

1.8 Mineral Reserve Estimate

A consolidated OP and UG mine plan was developed to extract the economic portions of KDM Indicated Mineral Resources plus stockpiled ore. The South Lobe is planned to be mined through a combination of OP and UG mining methods. Centre and North Lobe will not be further mined. Stockpiled ore consists of a mix of all three kimberlite lobes (South, Centre, North). All Mineral Reserves are classified as Probable.

The OP design and schedule was prepared by Lucara and reviewed by the QP and the UG design and schedule was prepared by QP. Stockpile quantities were prepared by Lucara and reviewed by the QP and are included in the Mineral Reserve Estimate. A consolidated summary of the Mineral Reserve Estimate, by mining method and lobe, is presented in Table 1-5.

The QP 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 September 30, 2025.

Table 1-5: Karowe Mine Mineral Reserve Estimate (September 30, 2025)

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	Price
		(Mt)	('000s ct)	(cpht)	(US$/ct)
Open Pit					
South - EM/PK(s)	Probable	0.1	25	21.0	695
South - M/PK(s)	Probable	1.0	109	10.8	520
Open Pit	**Total**	**1.1**	**134**	**11.9**	**553**
Underground					
South - EM/PK(s)	Probable	14.2	2,429	17.1	695
South - M/PK(s)	Probable	14.3	1,240	8.6	520
Underground	**Total**	**28.5**	**3,669**	**12.9**	**636**
Stockpile					
Mixed Stockpile	Probable	4.2	546	13.1	254
Life of Mine	Probable	3.5	198	5.7	442
Stockpile	**Total**	**7.6**	**745**	**9.8**	**304**
Combined					
All	**Total**	**37.3**	**4,547**	**12.2**	**579**

Notes:

1. Prepared by QP, 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;
4. The bottom elevation of the Probable Reserve is 250 masl;
5. Mineral Reserves are quoted above a +1.25 mm bottom cut-off diamond size and have been factored to account for diamond losses within the smaller sieve classes expected within the current configuration of the Karowe Mine Process Plant;
6. Average diamond price estimates are based on diamond sales data provided by Lucara Diamond Corp; prices were updated in 2025 for actual rough diamond sales over the period 2021 to end 2024, with partial sales data in 2025. The value of legacy diamonds (>US$5 million) has been incorporated in the financial model and are in addition to the diamond price estimate;
7. Tonnages are rounded to the nearest 100,000 tonnes, diamond grades are rounded to one decimal place to properly reflect the Reserve estimate accuracy. Rounding may cause minor discrepancies in the Mineral Reserve Estimate;
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 US$35/t based on an OP mining cost of US$13/t, a processing cost of US$12/t and a G&A cost of US$10/t;
10. Underground Mineral Reserves are estimated at a cut-off value of US$37/t based on an UG mining cost of US$15/t, a processing cost of US$12/t and a G&A cost of US$10/t;
11. OP dilution was assumed to be 0% as mining activities are primarily within the kimberlite. OP Mining Recovery of 95% has been assumed to account for potential challenges in achieving final bench elevations;
12. UG dilution assumptions were revised in 2025 as a result of geomechanical cave simulations. A total UG dilution of 23%, or 5.3 Mt has been included in the UG reserve estimate. Internal inferred domains are included as zero-grade waste. Cave drawdown simulations indicate that mine recovery will be 80% as of December 2037;

13. Stockpile Mineral Reserves are estimated at a cut-off value of US$19.50/t based on a rehandle cost of US$2.50/t, a processing cost of US$12/t and a G&A cost of US$5/t, when processed at the end of mine life;
14. Stockpile Reserves are not included in the Karowe 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. North lobe diamond price is estimated at US$155/ct; Centre lobe diamond price is estimated at US$210/ct.

1.9 Geotechnical and Hydrogeological Context

1.9.1 Geotechnical

Multi-year investigations (2017–2025) combining drilling, lab testing, photologging, hyperspectral analysis, and advanced modelling have defined the geotechnical conditions of the South Lobe pipe and host rocks. Above ~520 masl the pipe is bounded by sedimentary country rock (sandstones/mudstones) and below by granite basement. Rock mass indices and fracture frequency distributions show a general improvement in quality with depth into the basement. Portions of the Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup contain mudstone sub domains between 655 masl and 525 masl that locally degrade to “poor” while kimberlite and granitic basement units exhibit “good” rock mass quality overall supporting the placement of major underground excavations (crusher chambers, loading pockets, flood control levels, access drives) within the granite host unit.

Country rock (CR) mudstones and siltstones record the lowest Unconfined Compressive Strength (UCS) of 25–50 MPa, with other CR units typically 50–100 MPa (strong). Kimberlite facies average ~139 MPa UCS (very strong) with negligible intact strength anisotropy, supporting direction independent parameters in modelling. Reliable UCS and density–strength correlations for kimberlite underpin mine scale strength mapping.

A NW–SE fracture corridor terminates at the pipe and has been modelled along with minor structures to inform cave damage and propagation models.

The granite basement and kimberlite pipe, which form the primary focus of the Karowe underground mine plan, both demonstrate robust resistance to weathering under operational conditions. The upper portions of the granite basement exhibit some weathering, dominated by kaolinite alteration. These weathered zones are typically thin and localized, with most of the granite remaining fresh and competent at depth. Field and laboratory assessments confirm that the fresh granite is highly resistant to chemical and physical weathering, retaining its strength and integrity even after exposure. The kimberlite facies within the South Lobe show little to no alteration or degradation under repeated wet–dry cycles, indicating strong resistance to weathering. At the contact between kimberlite and country rock, some increased clay alteration (saponite, serpentinite) is present, but the bulk kimberlite remains structurally sound. Laboratory and field observations confirm that intact kimberlite maintains its rock mass strength even after exposure, supporting its suitability for long-term underground mining. Susceptible mudstones located above the granite basement can lose >50% rock mass strength on short exposure, necessitating prompt support and sealing of development faces.

The cavability of the South Lobe kimberlite was evaluated using a combination of empirical methods and advanced numerical modelling. Empirical assessments applied the Laubscher

IRMR/MRMR system and hydraulic radius calculations to map rock mass quality and geometry at various depths within the pipe. These analyses revealed that the upper portion of the kimberlite column, from approximately 665 masl to 480 masl, possesses favorable rock mass conditions and geometry for free caving, allowing gravity-driven cave propagation. In contrast, the lower section of the pipe, from about 480 masl to 310 masl, narrows significantly and exhibits less favorable structural characteristics, necessitating a transition to blast assisted cave mining to initiate and sustain cave development.

To validate and refine these empirical findings, large-strain coupled discontinuum and hydro-mechanical models were developed by Beck Engineering. These advanced simulations confirmed that free caving is achievable in the upper zones, while the lower zones require controlled drilling and blasting to create a broken, mobile rock mass. The models also highlighted the importance of pipe geometry, rock mass structure, and ambient stress conditions in controlling cave propagation and flow behavior. Notably, the transition from free to assisted caving was found to be depth-dependent, with the effectiveness of gravity-driven caving diminishing as the pipe narrows.

Overall, the combined empirical and numerical assessments support a dual-regime caving strategy for the UGP which involves free caving in the wider, structurally favorable upper column, and blast-assisted caving in the competent but less amenable lower column. Continuous monitoring and adaptive management are recommended to address potential risks, such as instability at the extraction level and variability in cave propagation, ensuring reliable and safe cave performance throughout the mine's life.

Extraction level footprint modelling suggests damage is non catastrophic but significant, specifying heavy ground support strategies and staged development to detect and mitigate emerging instability, especially near pipe edges, contact zones, and local domains of weakness.

Coarse resolution coupled modelling and site understanding indicate surface runoff, pit geometry, crater formation, fractured halo, and water bearing structures will channel water toward and then into the cave margins, with progressive migration into high flow draw columns as fines are mobilised. This elevates mudrush risk, choke potential, and wetting of muck. Mitigations include effective surface bunding, drainage management, pumped sumps, and an explicit mudrush risk management system incorporated into operations and trigger-action-response protocols (TARPs).

A comprehensive instrumentation and monitoring (I&M) program have been established to ensure the safe and effective development of the Karowe underground mine. The strategy integrates cave, surface, and excavation monitoring systems to provide real-time data on ground conditions, cave propagation, and operational performance. In the cave domain, smart markers and slough metres are installed in strategically placed boreholes to track three-dimensional cave movement and rockwall decoupling. A network of microseismic sensors, both underground and on the surface, enables detection of fracturing and blast effectiveness, particularly around critical drawbell pillars. Surface monitoring combines high-resolution deformation mapping, regular drone photogrammetry, and routine surveying to detect subsidence, pit break-back, and changes in surface infrastructure. Within underground excavations, LiDAR surveys and multi-point borehole extensometers are used to monitor deformation and stability in key areas such as the extraction and undercut levels, crusher chamber, and interburden zones. All monitoring data is centralized in an automated dashboard system, which incorporates TARPs to facilitate rapid operational decision-making. This integrated approach allows for adaptive mine management,

ensuring that emerging risks are identified early and that ground support and mining strategies can be adjusted in response to changing conditions.

1.9.2 Hydrogeological

The Karowe Diamond Mine employs a comprehensive groundwater management system for both its OP and UG operations. Water in the pit is managed primarily via a sump pump, with measured pumping rates increasing as the pit deepens. Dewatering boreholes (28 active as of 2025) are installed around the pit perimeter and within the pit to control inflow and depressurize slopes. Groundwater levels are monitored through a network of boreholes and piezometers, mainly in sandstone units, with limited data from deeper granite formations. Water quality monitoring indicates high salinity in deeper units, with total dissolved solids (TDS) reaching up to 46,000 mg/L in the granite and Mea formations.

The groundwater flow model was updated and calibrated in 2025 by Aqualogic (Aqualogic, 2025) and subsequently used to predict groundwater flow for future mine plans as presented herein. The model simulates pore pressure distributions and inflow rates over the LOM, using hydraulic conductivity (K) values established from field testwork.

The hydrogeology model incorporates mine plan data including:

- All existing and planned dewatering boreholes, with rates based on recent measurements from dewatering wells and underground probe holes;
- Updated OP and UG mine plans, including impermeable concrete-lined shafts; and
- Ground displacement due to caving operations.

Predicted Maximum Groundwater Inflows are 300 m^3/hr by 2032 and have been used to prepare underground water management plans and budgets.

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 2.85 Mt of kimberlite feed to the mill. All OP mining and processing activities are performed by Botswanan mine contractors working 365 days per year. The OP mine operation is expected to terminate mid-2026, ending at an elevation of approximately 665 masl.

There are substantial resources remaining below the economic extents of the OP that will be extracted by UG mine methods. A 7,800 t/d shaft operation utilizing fully assisted caving with transition to natural caving is under construction to provide an additional 10 years of mine life to the KDM operation after an eight-year construction period which commenced in 2020.

The UGP targets South Lobe resource, which is sub-cropping, vertical, roughly cylindrical, and narrows at depth. Two other Lobes, Center and North, have been mined out in the open pit, and any remaining mineralized material at depth is not classified as a resource nor pursued by underground mine methods.

The two Indicated domains of economic interest in the South Lobe are referred to as EM/PK(S) and M/PK(S). Several smaller domains (classified as Inferred) exist and will be incidentally mined and will, for reporting and economic modelling purposes, be treated as zero-grade dilution in the UG mine plan. The ratio of EM/PK(S) to M/PK(S) increases at depth and EM/PK(S) has approximately twice the diamond grade and contained value of M/PK(S). These two features drive several mine plan design decisions which focus on accessing the deeper, higher-value EM/PK(S) resource early in the mine life.

The large continuous nature of the South Lobe lends itself to bulk mine methods, and past studies have investigated several methods including sub level caving and blast assisted caving (BAC) / long hole shrinkage stoping (LHS), with the latter being featured in the 2019 Feasibility Study (Doerksen et. al., 2019) and the 2023 Feasibility Study (Doerksen et al., 2024). 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 below 500 masl. This feature led to the selection of LHS mine method which was planned to extract resources from 310 masl to the base of the OP.

Updates to geological, structural and geotechnical models and risk reviews conducted between 2024 and 2025 prompted a reassessment of natural caving scenarios to establish protocols for managing potential natural caving events at the mine. Hydraulically coupled cave simulations were performed to verify the location, progression rate, and development of a natural cave within the South Lobe. Findings confirmed that natural caving is not feasible at depth and predicts effective cave response 170 m above the proposed extraction horizon. This study recommends maintaining BAC mining up to the point where natural caving is expected, followed by a planned transition to block caving.

LHS stoping is planned to systematically drill and blast the base of the South Lobe using a vertical retreat approach. This method, conceptually similar to BAC, involves leaving the blasted muck in the excavation during stoping to provide host rock stability, with only the swell removed during the drilling and blasting phase. Mucking is performed from drawpoints located at a single extraction horizon 700 mbg. As ore is blasted, it expands beyond its original volume, and this additional material is extracted from the drawpoints to maintain an adequate void for further blasting within the cave. Once blasting progresses approximately 170 m above the extraction level, it is expected that the South Lobe will become unstable and initiate natural caving in 2029. LHS stoping will be sequenced strategically to control both the location and rate of cave back exposure, thereby managing the development of the natural cave. Following LHS stoping, mining operations will transition to conventional block caving, continuing draw from the drawpoints, with ongoing monitoring via instrumentation holes drilled from the surface and dedicated development drives. Cave draw will continue until dilution and recovery levels reach economic limits in 2037, some eight (8) years following cave initiation. Once cave draw concludes the underground mine will begin closure and reclamation processes while processing remaining OP stockpile inventories.

Access to the UG mine will be from a 767 m deep Production Shaft (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 personnel and material movement, and an auxiliary cage for shaft inspections and personnel transport. Shaft conveyances will be managed by three independently operated winders. This shaft will also serve as the main fresh air intake to the mine. A second Ventilation Shaft (V/S), 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

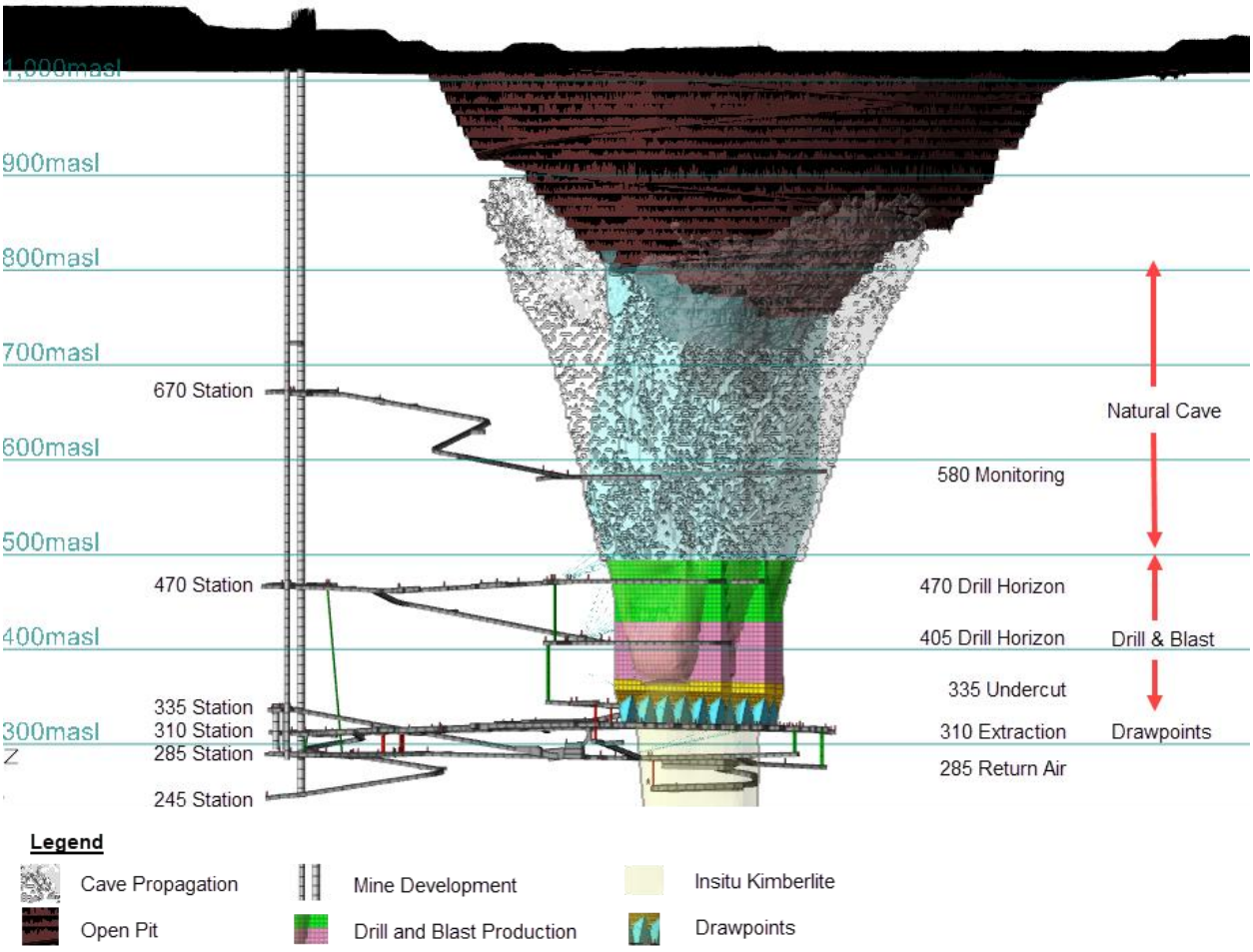
zone, and 100 m from each other. Shafts have been sunk blind using conventional drill and blast equipment and have been developed concurrently. As of the report date the P/S is sunk and lined to final elevation and entering final equipping stage, and the V/S is sunk to its final elevation. Shaft sinking construction efforts commenced in 2020 and are expected to conclude in 2026.

It is expected to take approximately two years from shaft handover 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 or mining level (lvl). The 310 lvl 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 three drilling horizons: 340 lvl, 405 lvl, and 470 lvl; 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 (looking North)



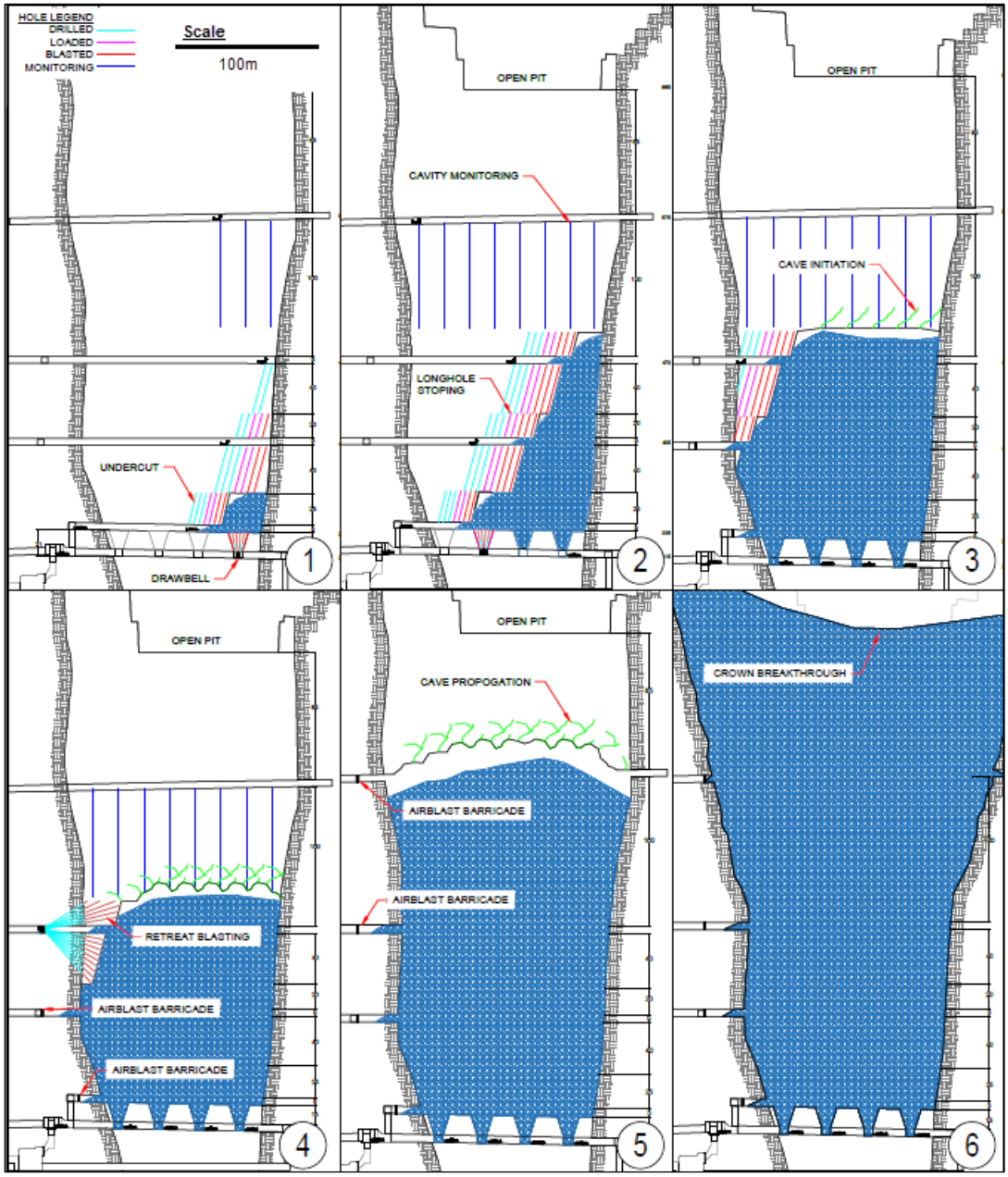
The UG lateral development will be driven by seven bolt-bore development jumbos. The jumbos will be mobilized in a phased approach with most of the early development occurring on the 285 lvl and 310 lvl. During the peak lateral development stage, each jumbo will drive an average of 3.75 m/day. Heading advance rates will range from 1.3 – 4.5 m/day depending on the heading profile size and ground support requirements. During pre-production, a total of 16 km of development will be driven.

Drill horizons are spaced at 65 m vertical intervals. Stopes will be mined in 65.0 m x 31.5 x 5.0 m rings in retreat fashion using both downholes and up holes. Two stoping levels, each with four parallel stopes, will be mined from east to west in an en echelon pattern to target a stable arched cave back profile. Dump angles of 20 degrees and ring offsets between blasting horizons using both down and up hole drilling will achieve a toe-to-toe cave front of approximately 50 degrees.

Six In-the-hole hammer (ITH) drills will be utilized to drill and blast approximately 21,000 t/d in order to supply 7,500 t/d of swell to the draw bells for the first two years of operation.

Figure 1-3 illustrates a schematic cross section of the pipe, showing the advance of stopes and transition to natural cave.

Figure 1-3: Mining Method Illustration



Notes:
Looking North

A single El Teniente style extraction level will be made up of five panels that are driven 31.5 m apart and run the entire length of the lobe. Panels will access one of 29 drawbells which are required for the life of mine. A perimeter drive around connecting extraction panels 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 maintenance of the comminution circuit and a quick re-handle once the circuit returns to normal operation. Four loaders, two 21 t and two 17 t, initially used for lateral development will be required to maintain production at the draw bells. As loaders are replaced over the mine life, opportunities for standardizing 21 t fleets or moving towards hybrid or electric equipment will be investigated.

Material dumped onto the grizzly will feed a 1.3 m x 1.5 m (50” x 60”) UG jaw crusher with 643 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 lvl shaft station and discharged into one of two 7.3 m diameter fine ore storage bins, each with a capacity of 2,600 t at 75% fill.

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 1-6 states the annual schedule of material hoisted to surface from the UG operation.

Table 1-6: UG Production Schedule

Year	EM/PK(S)			M/PK(S)			Total		
	Tonnes (1)	Grade	Carats	Tonnes (1)	Grade	Carats	Tonnes (1)	Grade	Carats
	Mt	cpht	Kc	Mt	cpht	kc	Mt	cpht	kc
2026	0.0	15.9	2	0.0	0.1	0	0.01	15.0	2
2027	0.5	15.4	75	0.0	6.5	2	0.52	14.9	77
2028	2.3	17.9	411	0.2	8.8	13	2.45	17.4	425
2029	2.5	18.5	460	0.3	9.6	29	2.78	17.6	489
2030	2.2	19.7	431	0.6	10.2	62	2.79	17.6	493
2031	1.4	20.5	296	1.4	10.5	148	2.85	15.6	444
2032	0.9	20.4	175	2.0	10.5	211	2.86	13.5	385
2033	0.6	20.3	125	2.2	10.4	234	2.85	12.6	358
2034	0.6	18.9	122	2.2	9.6	213	2.85	11.7	334
2035	0.8	15.4	118	2.1	8.0	166	2.85	10.0	284
2036	1.0	10.9	114	1.8	5.7	104	2.86	7.6	218

Year	EM/PK(S)			M/PK(S)			Total		
	Tonnes (1)	Grade	Carats	Tonnes (1)	Grade	Carats	Tonnes (1)	Grade	Carats
	Mt	cpht	Kc	Mt	cpht	kc	Mt	cpht	kc
2037	1.3	7.6	101	1.5	4.0	60	2.85	5.6	161
Total	**14.2**	**17.1**	**2,429**	**14.3**	**8.6**	**1,240**	**28.5**	**12.9**	**3,669**

Notes:
1. Tonnes are fully diluted with waste and inferred dilution indirectly mined from drawpoints with ore.

The ventilation network will consist of 422 cubic metre per second (m^3/s) fresh air intake by the P/S and one exhaust route by the V/S. Fresh air will be pulled into the mine workings through the P/S. Bifurcated surface exhaust fans will be mounted over the V/S collar to service the mine. 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) through a seasonally operated 7.5 Mega Watts of Refrigeration (MWR) surface bulk air cooler plant mounted to a subsurface plenum connected to the P/S fresh air intake.

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 will be constructed in phases with a final configuration of 16 pumps operating in parallel, with a peak dewatering capacity of 550 cubic metres per hour (m^3/hr) during stormwater events. 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 will peak in 2027 with on-site dayshift requirements of 553 and 955 total employed.

The OP will continue to operate until Q2 2026. 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 1-4 through Figure 1-6.

Figure 1-4: Summary of Mine Production

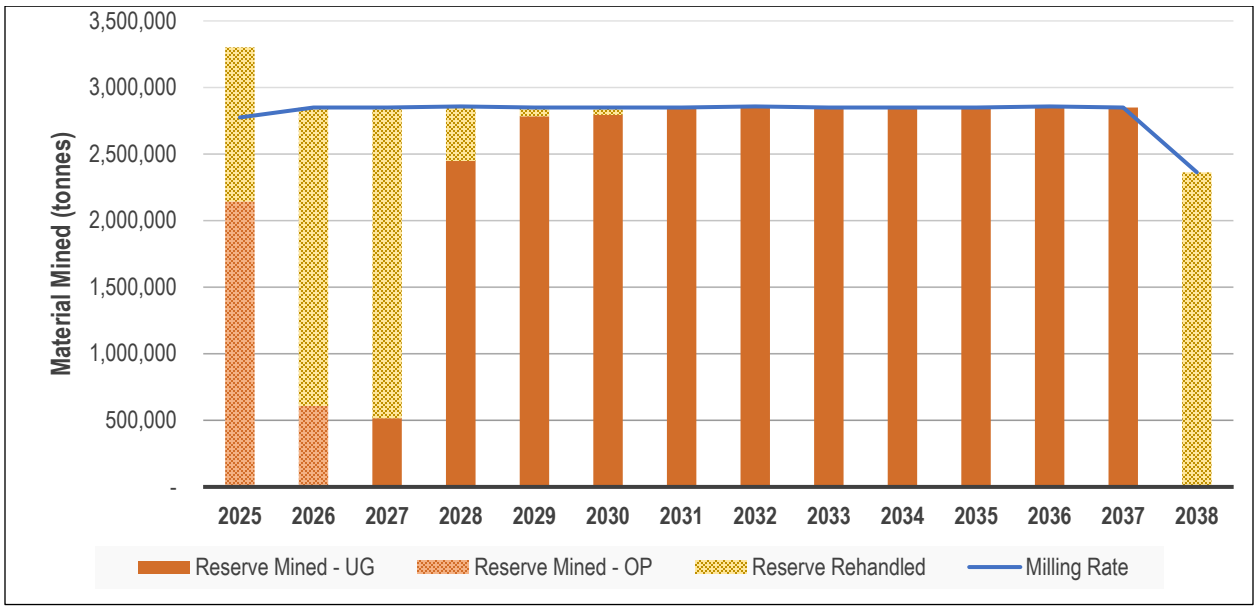
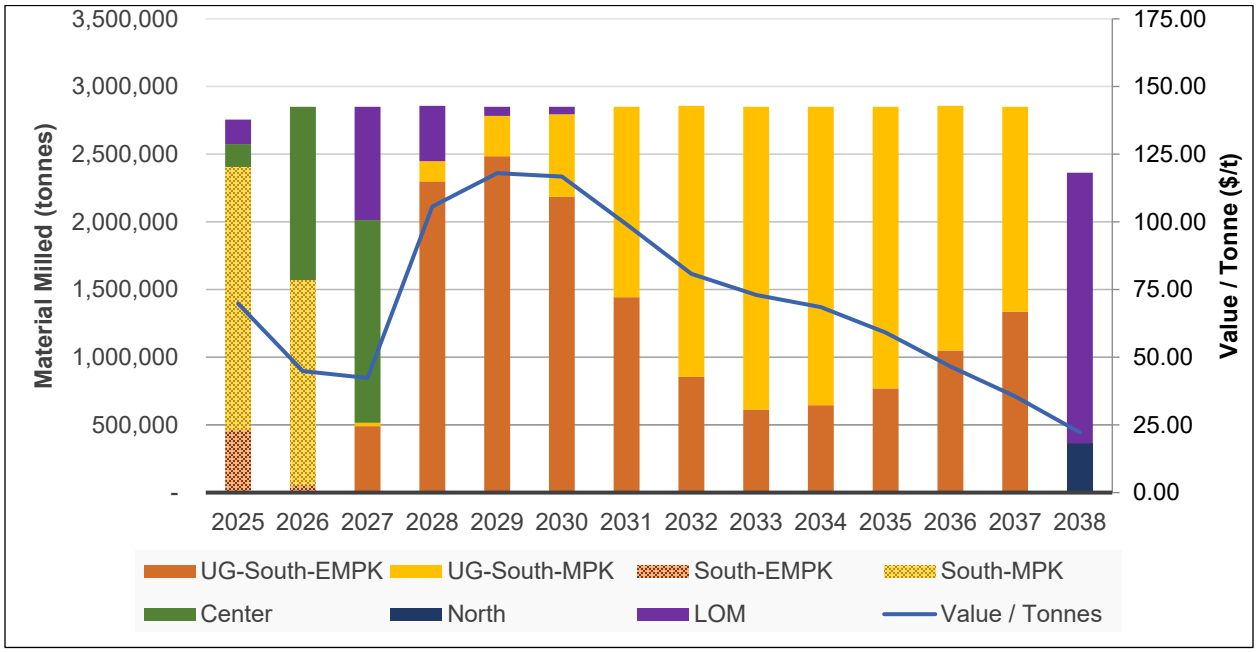
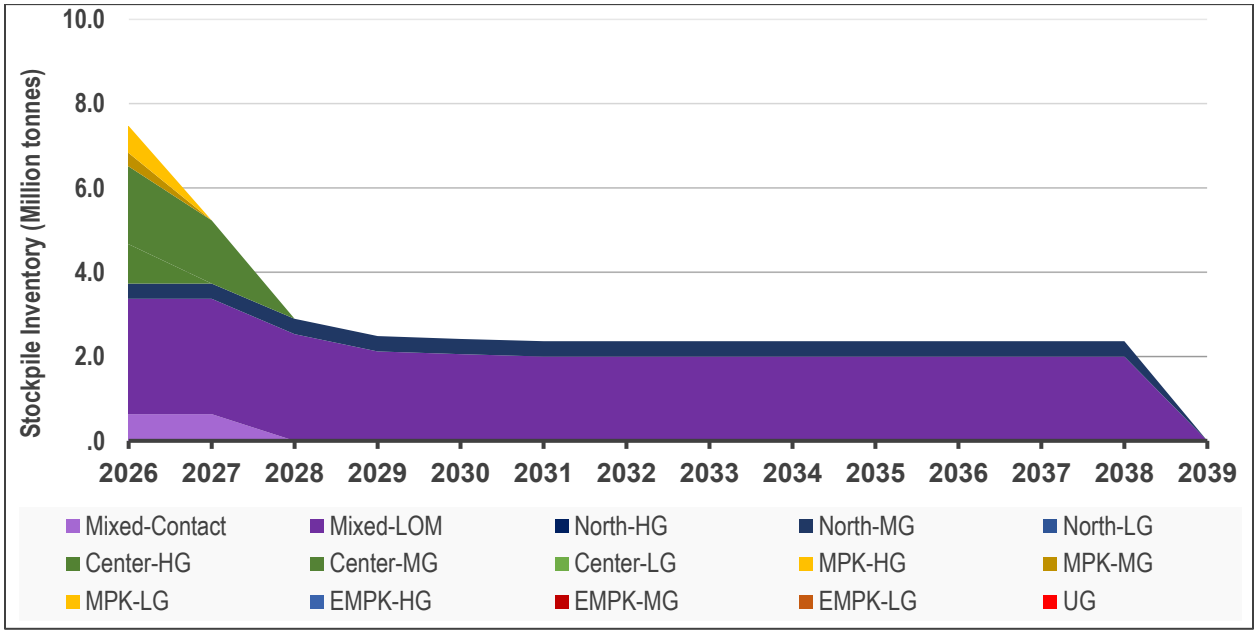


Figure 1-5: Summary of Mill Production



A summary of the stockpile inventory opening balance is outlined in Figure 1-6.

Figure 1-6: Summary of Stockpile Inventory Opening Balance



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) and 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 magnetic roll (or MagRoll) separators and X-ray recovery system for material volume 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 in addition to 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 at Karowe will utilize a combination of existing and newly constructed infrastructure at the Karowe Mine. All current and planned surface installations have been designed to fully support the long-term operation of the underground mine and the processing plant. Over the past

four years, Project construction has advanced with the completion of the majority of key surface infrastructure components. These include:

- A new 132 kV substation and switchyard at Botswana Power Corporation's Letlhakane substation, 29 km-long 132 kV overhead powerline to site, on site 11 kV distribution to the UGP E-house, and eight MW of diesel generator back-up power;
- Expansions to water treatment plants and distribution of filtered and potable water;
- Surface water management ponds for settling and mechanical evaporation;
- Construction facilities including a 200 person camp, office complex, warehouse, change house, lamp room, material testing, laydowns, and workshops;
- All required sinking headframes, hoist houses, and associated infrastructure;
- Fully sunk and lined shafts with shaft stations excavated;
- All required permanent shaft winding equipment and buildings;
- All permanent fixed plant infrastructure including compressors, bulk air coolers, grout and batch plants; and
- Surface fire water systems, hydrants, and detection systems.

This completed infrastructure forms the foundation for the transition to underground mining and positions the Project for the safe and efficient execution of the remaining development activities.

The UGP will make use of existing operation infrastructure including the processing plant, site access road, airstrip, pit dewatering pipeline, maintenance facility, Fines Residue Disposal (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 primary ventilation fans, permanent P/S conveyances, UG control room, and sustaining water management ponds and TSF expansions.

1.12.1 Tailings Management

Knight Piésold (KP) conducted a 2019 Feasibility Study to address evolving operational needs and environmental considerations for the design requirements of FRD 1. The study initially proposed raising FRD 1 to 1,042 masl and constructing a new FRD 2 at the same elevation. However, 2021 design revision imposed a height restriction on FRD 1, capping it at 1,031 masl, while FRD 2 was redesigned to store tailings until 2027 within existing boundaries.

Construction of FRD 2 began in 2022, featuring two paddocks separated by a wall and using a two-stage lifting process. A 2022 site selection study identified a new location for FRD 3 west of the existing facilities, with detailed designs concluded in 2024. Both FRD 2 and FRD 3 designs adhere to a final design criteria elevation of 1,031 masl, in line with the 2024 FRD 3 feasibility

design requirements. As deposition into FRD 2 proceeds according to underground production plans, this Report details the feasibility designs for the Coarse Residue Deposit (CRD), FRD 2, and FRD 3, outlining a strategic approach for sustainable mine residue storage.

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 Protection (formerly Department of Environmental Affairs) during the same year.

A new EMP has also been approved in 2024 for the 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. By 2031, additional produced water disposal plans will need to be developed for the remaining LOM. The first of three ponds was constructed and licensed for operation in 2024. Future ponds will 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 July 2025, and the next review is scheduled for July 2027. Lucara's 2021 Human Rights Review and Human Rights Risk Assessment in 2024 (Prizma, 2025) identified access to water as a salient topic.

The area of the Mining Licence 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*, International Union for Conservation of Nature (IUCN): 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" and considered threatened. KDM enacted a Memorandum of Understanding (MOU) with Birdlife Botswana in January 2023 and with their assistance conducted a comprehensive Biodiversity Assessment Survey of the KDM lease area in July 2024.

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. In 2022 surveys were conducted for FRD 2 and the underground water management pond was surveyed in March 2025. No sites of archeological or cultural importance were identified in these surveys, and the project area remains graded "5" (No further archaeological work required) on the Botswana Department of National Museum and Monuments (DNMM) grading scale. Monitoring will continue again in future during construction of FRD 3.

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 functions.

KDM received ISO 45001 recertification for its occupational, health and safety management system in November 2024, and is pursuing ISO 14001 certification for its environmental management system. Lucara is also a certified Member of the Responsible Jewellery Council (expires March 2027) and is a Participant of the United Nations Global Compact (UNGC) and completed the 2025 Communication on Progress in 2025.

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 (GHG) emissions, waste management, biodiversity, environmental incidents, and community grievances. The results are reported to regulators, project financiers, and other stakeholders, including via Lucara's web published annual sustainability reports (latest publication was for the 2024 period).

KDM is connected to the national grid and operates a diesel-fueled mobile fleet. Lucara continues to publicly disclose its annual GHG emissions and plans. These describe a series of decarbonization initiatives, a completed feasibility study for a large-scale solar PV power plant, and opportunities to reduce GHG emissions. Lucara's 2024 reporting indicates that KDM's GHG emissions totalled 96,961 tonnes of carbon dioxide equivalent (tCO_{2e}) (Scope 1 and 2) and GHG intensity was 24.57 (Total CO_{2e}(kt)/ore + waste rock mined (t)).

As part of the EMP, a Mine Closure and Rehabilitation Plan (MCRP) and associated costing was updated in 2024. The estimated reclamation liability is approximately US$37.9 million. Lucara Botswana has provided financial guarantees totalling BWP 270.0 million for reclamation obligations.

1.14 Capital and Operating Costs

1.14.1 Capital Cost Estimate

The capital cost estimate consists of three parts:

- Pre-production period - Incurred – UGP capital costs prior to the effective date of the FS (September 30, 2025) of US$436.0 million;
- Pre-production period – To Completion – Estimated UGP capital costs from the effective date of the FS until commercial production of US$343.2 million; and
- Sustaining – Estimated UGP capital costs occurring after commercial production (March 1st, 2028), as well as capital costs for existing infrastructure constructed prior to the UGP, totalling US$240.9 million.

Total UGP specific pre-production capital costs (incurred plus to completion), are estimated to be US$779.2 million.

The cost estimate is a Class 3 Estimate according to the Association for the Advancement of Cost Engineering (AACE).

Costs are expressed in 2025 US$ with no escalation unless stated otherwise.

Table 1-7 outlines the capital cost estimate.

Table 1-7: Summary of Capital Cost Estimate for LOM

WBS	Capital Costs	Pre-Production			Sustaining (US$M)	LOM Total (US$M)	Weight (%)
		Incurred (US$M)	To Completion (US$M)	Subtotal (US$M)			
1000	Mining	254.6	183.9	438.5	59.4	497.9	53
2000	Site Development	17.6	3.8	21.4	6.8	28.2	3
3000	Process Plant	-	-	-	-	-	0
4000	Tailings and Mine Waste	-	-	-	42.4	42.4	5
5000	On-site Infrastructure	11.6	0.2	11.8	92.2	104.0	11
6000	Buildings and Facilities	2.5	0.7	3.3	-	3.3	0
7000	Off-site Infrastructure	27.0	-	27.0	-	27.0	3
8000	Project Indirects	15.9	10.6	26.5	2.6	29.1	3
9000	Owner Costs	106.8	91.6	198.4	0.8	199.2	21
Subtotal		**436.0**	**290.8**	**726.8**	**204.1**	**930.9**	**100**
10000	Contingency	-	52.4	52.4	7.0	59.3	
11000	Closure	-	-	-	29.9	29.9	
Total Capital Costs		**436.0**	**343.2**	**779.2**	**240.9**	**1020.1**	

Notes:
*Numbers may not add due to rounding.

1.14.1.1 Mining

Mining capital costs consist of primarily UG mining activities; there are no capital costs associated with OP operations. Incurred costs include UGP surface and UG infrastructure that have been constructed prior to the effective date of this Report. Capital costs to completion are primarily associated with UG development and infrastructure costs, as well as the sustaining costs associated with UGP.

UG Mining capital costs contain a mix of actual costs, purchase orders, budgetary quotes from local vendors, and database costs from comparable projects. Time related costs for development or infrastructure installations include third party vendor and contractor estimates. Contractor labour rates are based on the existing site contracts for shaft sinking and lateral development, while KDM provided Owner's Team salaries are used for operations phase rates.

Table 1-8 provides a mining capital cost breakdown.

Table 1-8: Mining Capital Costs

WBS	UG Mining Capital Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1100	Surface Infrastructure	59.4	6.2	-	65.6	13
1200	Shaft Sinking and Infrastructure	182.9	37.1	-	219.9	44
1300	UG Development	-	60.8	16.2	76.9	15
1400	UG Equipment	5.9	36.0	35.3	77.2	16
1500	UG Infrastructure	6.4	29.2	7.9	43.5	9
1600	Capitalized UG Operating Costs	-	14.6	-	14.6	3
1000	**Total Mining**	**254.6**	**183.9**	**59.4**	**497.9**	**100**

1.14.1.2 Site Development

Pre-Production surface earthworks and civils costs associated with the UGP are complete, inclusive of geotechnical programs. The surface dewatering pond and initial phase of mechanical evaporators is substantially complete, with remaining costs and pond expansions scheduled to occur in the sustaining period. A hydrological kimberlite depressurization drilling program is planned prior to developing into the orebody.

Table 1-9: Site Development Capital Costs

WBS	Site Development	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
2100	Bulk Earthworks	11.8	-	-	11.8	42
2200	Site Roads	-	-	-	-	0
2300	Surface Water Management	3.2	0.6	6.8	10.6	38
2400	Dewatering	-	3.2	-	3.2	11
2500	Core Hole Drilling	2.5	-	-	2.5	9
2000	**Total Site Development**	**17.6**	**3.8**	**6.8**	**28.2**	**100**

1.14.1.3 Tailings Facilities

Tailings facility costs are associated with the establishment of new facilities or major expansions of the existing facilities on site. Costs associated with routine projects on the FRD are capitalized and covered under the existing on-site infrastructure budget.

Sustaining costs for tailings consist of an additional lift to FRD 2 where tailings deposition is currently underway, as well as an expansion of FRD 3. All tailings facility costs are considered sustaining capital, as costs scheduled prior to completion of the UGP in Q1 2028 are primarily associated with ongoing open pit operations and the processing of existing stockpiles.

While both the CRD and the WRSF will be expanded as part of the UGP, these activities are minor and captured under operating budgets.

Table 1-10: Tailings Facility Costs

WBS	Tailings Facility	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
4100	FRD - Fines	-	-	42.4	42.4	100
4200	CRD - Coarse	-	-	-	-	0
4300	Waste Rock Storage Facility	-	-	-	-	0
4000	**Total Tailings**	**-**	**-**	**42.4**	**42.4**	**100**

1.14.1.4 On-Site Infrastructure

All on-site infrastructure such as electrical power and service water supply and distribution projects to support the UGP are substantially complete. Sustaining capital costs for on-site infrastructure are associated with maintaining infrastructure constructed prior to the UGP. This includes facilities such as the process plant, tailings facility, site security system, site services, and access road. All existing on-site infrastructure costs are considered sustaining capital, as costs scheduled prior to completion of the UGP in Q1 2028 are primarily associated with ongoing open pit operations and the processing of existing stockpiles.

Table 1-11: On-site Infrastructure Costs

WBS	On-Site Infrastructure	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
5000	Existing On-site Infrastructure	-	-	92.2	92.2	89
5100	Electrical Supply and Distribution	8.8	0.2	-	9.0	9
5200	Water Supply, Distribution, and Treatment	2.3	-	-	2.4	2

WBS	On-Site Infrastructure	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
5300	Waste Collection and Treatment	0.5	-	-	0.5	0
5400	IT and Communications	-	-	-	-	0
5000	**Total On-site Infrastructure**	**11.6**	**0.2**	**92.2**	**104.0**	**100**

1.14.1.5 Buildings and Facilities

Buildings and facilities on site are established with remaining costs associated with additional mine rescue equipment requirements as underground development and construction activities progress.

Table 1-12: Buildings and Facilities Costs

WBS	Building and Facilities	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
6100	Training Center	-	-	-	-	0
6200	Workshop and Warehouse	0.2	-	-	0.2	5
6300	Mine Rescue Centre	1.7	0.7	-	2.4	73
6400	Offices	0.5	-	-	0.5	15
6500	Change house	0.2	-	-	0.2	6
6600	Access, Fencing, and Traffic Management	-	-	-	-	0
6000	**Total Buildings and Facilities**	**2.5**	**0.7**	**-**	**3.3**	**100**

1.14.1.6 Off-site Infrastructure

Off-site infrastructure construction is complete including the construction of the new BPC electrical transmission line and associated substations, along with the construction of the contractor's camp.

Table 1-13: Off-site Development Costs

WBS	Off-Site Development	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
7100	Power Transmission Line	22.3	-	-	22.3	83
7200	Off-site Accommodations	4.7	-	-	4.7	17
7000	**Total Off-site Development**	**27.0**	**-**	**-**	**27.0**	**100**

1.14.1.7 Project Indirect Costs

Project indirects cover camp catering, office rentals, bussing, material haulage, civil material testing, freight and freight forwarding services, and flights for personnel. After commercial production in Q1 2028, project indirect costs associated with UG contractors completing remaining scope has been capitalized, while project indirects for the Owner's Team are considered operating costs.

Table 1-14: Project Indirects Costs

WBS	Project Indirects	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
8100	On-site Contract Services	8.0	8.2	2.6	18.8	65
8200	Temporary Facilities and Utilities	-	-	-	-	0
8300	Contractor Indirects	0.3	0.6	-	0.9	3
8400	Freight	6.8	0.5	-	7.4	25
8500	Temporary Accommodations and Expenses	0.8	1.3	-	2.0	7
8000	**Total Project Indirects**	**15.9**	**10.6**	**2.6**	**29.1**	**100**

1.14.1.8 Owner's Costs

Owner's costs are classified as the management, oversight, and site operation costs that are instrumental to develop and operate the UGP. These costs are capitalized during the construction phase and treated as an operating cost following commercial production. Only specific free-issued materials such as ground support, concrete and shotcrete associated with contractor development and construction are capitalized in the sustaining period.

Owner's costs include:

- Owners labour;
- 3rd party engineering and consulting services;
- Free-issue materials including fuel, power, explosives, ground support, concrete, and shotcrete;
- Project taxes and insurance;
- Safety and Training;
- Site Services;
- Equipment fleet maintenance; and
- Fixed equipment maintenance (shafts, BAC, crusher and conveyor, etc.) and major component replacements.

The Project's external EPCM team transitioned to roles within Lucara at the end of Q1 2025 and is now fully integrated within the Owner's team and labour structure.

Table 1-15: Owner's Costs

WBS	Owners Capital Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
9100	Pre-Production General and Administration	14.6	41.2	-	55.8	28
9200	Operational Charges	15.9	16.0	-	32.0	16
9300	Engineering, Procurement, and Construction Management	56.7	4.7	-	61.4	31
9400	Equipment Supply and Maintain	0.9	2.0	-	2.9	1
9500	Free-Issue Materials	18.6	27.7	0.8	47.2	24
9000	**Total Owners Costs**	**106.8**	**91.6**	**0.8**	**199.2**	**100**

1.14.1.9 Contingency

Project contingency costs consist of three categories:

- Owners – General project contingency of approximately 10% of the estimated cost to completion;

- Project Change Log – A Project Change Log has been developed which outlines specific potential changes to existing project scope or contracts. These costs are not planned and not contractual but are expected to be incurred based on historical project performance; and

- Sustaining – General project contingency of approximately 10% of the estimated sustaining capital costs at the underground working area. This excludes sustaining capital costs on the existing on-site infrastructure and tailings facilities, as these costs are more defined and based on historical budgets.

No provision is made, or contingency allowed for external project risk, unidentified changes to the scope or work, general project escalation, or foreign exchange fluctuations. Limited contingency allowance for internal project risk has been allowed as defined in the Project Change Log. Remaining pre-production contingency is approximately 15% of the remaining cost to completion.

Table 1-16: Contingency

WBS	Contingency	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
10110	Owners	-	30.0	-	30.0	50
10130	Project Change Log	-	22.4	-	22.4	38
10140	Sustaining Capital Contingency	-	-	7.0	7.0	12
10000	**Total**	**-**	**52.4**	**7.0**	**59.3**	**100**

1.14.1.10 Closure

Closure costs encompass the entire KDM site inclusive of the OP, UGP, tailings facilities, process plant and all related infrastructure.

Lucara Botswana has provided financial guarantees totalling BWP 270.0 million for reclamation obligations, consisting of a restricted deposit of BWP 70.0 million and a BWP 200 million standby letter of credit (First National Bank of Botswana Limited, 2025). The closure costs have been estimated at US$29.9 million, which is an estimated total cost of US$34.9 million less the restricted deposit of US$5 million (BWP 70.0 million), excluding contingencies.

Table 1-17: Closure Costs

WBS	Closure Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
11110	Pit Buildings and Surface	-	-	2.8	2.8	9
11120	Open Pit and Dumps	-	-	11.4	11.4	38
11130	Slimes and Dams	-	-	10.3	10.3	34
11140	UG	-	-	1.4	1.4	5
11150	Monitoring	-	-	2.1	2.1	7
11160	Project Management	-	-	1.9	1.9	6
11000	**Total Closure Costs**	**-**	**-**	**29.9**	**29.9**	**100**

1.14.2 Operating Cost Estimate

Operating costs include on-site mining, processing, coarse and fine residue deposition, and administration costs. Off-site, in-country corporate costs such as Lucara Botswana management and cost of sales have been provided by Lucara and are included as sales and corporate costs. Out of country corporate costs associated with Lucara Diamond have been excluded from the operating costs.

All UGP costs incurred during the construction phase are capitalized and form part of the capital cost estimate. All other KDM operating costs incurred during the construction phase to support the current operations are included as part of operating costs.

All costs are in 2025 US dollars (US$ or $), with no considerations for escalation or inflation.

Total operating costs over the LOM are US$1,315 million. A summary of operating costs for the site is provided in Table 1-18.

Table 1-18: Summary of Operating Cost Estimate

Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Open Pit Mining Costs [3]	10.2	20.3	1.1	18.0	2
UG Mining Costs	42.7	427.3	28.5	15.0	32
Stockpile Rehandle Costs	3.8	19.3	7.6	2.5	1

Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Process Costs	24.7	345.3	37.3	9.3	26
Other Power Costs	7.1	99.2	37.3	2.7	8
G&A	18.5	259.1	37.3	6.9	20
Cost of Sales	3.6	49.8	37.3	1.3	4
Corporate Charges (Botswana)	6.8	95.0	37.3	2.5	7
Total	**94.0**	**1,315.4**	**37.3**	**35.3**	**100**

Notes:
1. Average cost per year in which costs occur.
2. Tonnes processed in relation to operating cost category.
3. Includes rehandle costs incurred during open pit operations and demobilization costs as the end of open pit mine life.

The main assumptions used to build up the operating costs are in Table 1-19.

Table 1-19: Operating Cost Assumptions

Item	Units	Base	Source
Exchange Rates, Escalation, and Taxes			
South African Rand	xZAR:1USD	17.50	Historical Average
Botswana Pula	xBWP:1USD	14.00	KDM
Escalation Rate	%	0	Excluded
Value Added Tax (VAT)	%	14	BURS
Power			
Fixed Charge	BWP/month	115.05	BPC Line Power Delivered to site excluding VAT
Demand Rate	BWP/kW	312.44	
Energy Charge	BWP/kWh	1.07	
Fuel			
Diesel Fuel, 50 ppm	BWP/L	15.11	September 15th, 2025 BERA update Petrohyper Delivered to site excluding VAT
Labour			
A2	BWP/month	15,409	2025 budgets mid-range fully burdened 5x2 shift base case
B1	BWP/month	18,988	
B2	BWP/month	21,282	
B3	BWP/month	25,246	

Item	Units	Base	Source
B4	BWP/month	29,664	
C1	BWP/month	41,783	
C2	BWP/month	51,375	
C3	BWP/month	64,862	
C4	BWP/month	71,619	
D1	BWP/month	95,782	
D2	BWP/month	114,634	
D3	BWP/month	140,469	
D4	BWP/month	160,950	

As the KDM is currently in operation, the operating cost estimates for OP mining, processing and other power, G&A, cost of sales, and corporate charges, were prepared using forecast operating budgets provided by Lucara. The UG mining operating costs were prepared using first principles, and budgetary quotations.

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. No contingency has been applied to UG mining operating costs.

The total LOM operating costs for the UG Mining operations are summarized in Table 1-20.

Table 1-20: UG Mining Operating Cost Summary by Activity

Operating Costs	Average Annual [1]	Life of Mine	Tonnes Processed [2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Lateral Development [3]	0.3	2.6	28.5	0.09	1
Drill & Blast	11.7	23.4	28.5	0.82	5
Load and Haul	2.4	23.7	28.5	0.83	6
Crushing and Hoisting	6.2	62.3	28.5	2.18	15
Surface Haulage	6.0	60.3	28.5	2.11	14
Mine Maintenance	10.6	106.0	28.5	3.72	25
Mine General	14.9	149.1	28.5	5.23	35
Total	**42.7**	**427.3**	**28.5**	**14.98**	**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;
2. Tonnes processed in relation to underground operating cost; and

3. Lateral Development costs are primarily performed by a contractor and capitalized. Costs in this activity are the standard free-issued materials (explosives, fuel, power) provided by the Owner's Team for both lateral development and rehabilitation activities occurring throughout the mine life.

1.15 Economic Analysis

An after-tax economic model was developed to estimate the annual cash flows and sensitivities of the Project for KDM. This Report does not consider the UGP as a stand-alone asset nor evaluates it as a stand-alone economic cash flow. The summaries provided represent the LOM outputs, which include the remaining OP, current stockpiles, and the additional value from the development of the UGP.

The summary of the mine plan, revenues and costs over the LOM is outlined in Table 1-21.

Table 1-21: LOM Summary of Economic Results

Parameter	Unit	Value
Ore Processed	Mt	37.3
Mill Average Annual Production	Mt	2.85
Average Processing Grade	cpht	12.2
Diamonds Recovered	k ct	4,547
Recovery	%	100.0
Net Revenue (after royalties)	US$M	2,991.9
Operating Costs	US$M	1315.4
Income Taxes and Working Capital	US$M	316.5
Pre-Production Capital Cost (excluding incurred costs to September 30, 2025)	US$M	290.8
Sustaining Capital Cost and Closure	US$M	234.0
Contingency	US$M	59.3
After-tax Free Cash Flow	US$M	775.9
Net Present Value @ 8%	US$M	432.1

1.15.1 Assumptions and Economic Factors

Table 1-22 and Table 1-23 outlines the diamond price 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 and sales, as outlined in Section 19.

Table 1-22: Baseline Diamond Prices

Unit	Unit	2025 FS
North	US$/ct	155
Centre	US$/ct	210
EM/PK(S)	US$/ct	695
M/PK(S)	US$/ct	520
Stockpiles	US$/ct	442

Table 1-23: Legacy Stone (>US$5 million) Values

Unit	Occurrence Stones / Mtonne	Diamond Price US$M / Stone	Revenue US$M / Mtonne [1]
EM/PK(S)	3.34	12.9	43.0
M/PK(S)	1.45	9.5	13.7

Notes:
1. Undiluted tonnes.

Other key assumptions and economic factors include the following:

- Discount rate of 8%;
- Nominal 2025 US$ dollars;
- Foreign exchange rates of BWP14.00:US$1.00 and ZAR17.50:US$1.00 are used where applicable;
- Revenues, costs, and 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;
- Lucara Botswana corporate costs are included in all economic results;
- Canada corporate (Lucara Diamond Corp.) costs are excluded in the base case economic model results;
- Working capital is included;
- Debt financing costs are excluded; and

- The model excludes all incurred costs up to the base date of September 30, 2025.

After-tax estimates of project values were developed and are likely to approximate the true investment value. It must be noted, 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.

The economic estimates in this 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. The economic model was prepared by Lucara Diamond and verified by the QP. Lucara Diamond has performed the tax estimates based on their >10 years of operating experience in Botswana.

This Report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. No provision is made, or contingency allowed for external project risk, unidentified changes to the scope or work, general project escalation, or foreign exchange fluctuations, some of which may cause actual results to differ materially from those presented. Limited contingency allowance for internal project risk has been allowed as defined in the Project Change Log. A summary of risks that apply to Lucara Diamond, KDM, and the UGP are described in Section 25.2.

The reader is cautioned that diamond price and legacy stone valuations are the primary drivers in the economic model. Efforts have been made to provide realistic estimates for diamond prices based on historical performance and sales information, but it should be noted that diamond prices are based on many complex factors and there are no reliable long-term predictive tools and no guarantee that they will be realized in the future.

1.15.2 Results

The KDM LOM, including the development of the UGP, has an after-tax net present value using an 8% discount rate ($NPV_{8\%}$) of $432.1 million, using the cost assumptions described in Section 1.15.1. A summary of the after-tax economic results for the Project, based on the assumptions outlined above, are presented in Table 1-24.

Table 1-24: After-Tax Economic Results - LOM Model

Parameter	Unit	After-tax Results
$NPV_{8\%}$ excluding Canadian corporate costs	US$M	432.1
$NPV_{6\%}$ excluding Canadian corporate costs	US$M	505.5
$NPV_{8\%}$ including Canadian corporate costs	US$M	368.5
$NPV_{6\%}$ including Canadian corporate costs	US$M	434.7

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

The after-tax break-even diamond price for the Project (US$0 NPV @ 8% discount rate) is 67% of the assumed diamond price and legacy diamond value.

1.15.3 Sensitivities

A univariate sensitivity analysis was performed using diamond prices and mine costs as variables. The value of each variable was tested using the same percentage range of variation, from -20% to +20%, independently while all other variables were held constant.

The Project is most sensitive to the diamond price and least sensitive to capital costs. The results of the sensitivity analysis are shown in Table 1-25.

Table 1-25: Sensitivity Results After-Tax (NPV @ 8%)

Variable	After-tax $NPV_{8\%}$ (US$M)				
	-20% Variance	-10% Variance	Base	+10% Variance	+20% Variance
Diamond Price	248.8	342.6	432.1	518.0	602.4
Mining Cost	457.5	445.0		418.8	404.7
Processing Cost	454.6	443.6		420.6	408.8
All Operating Costs	512.1	473.1		388.0	342.2
Upfront CAPEX	474.3	453.2		411.3	390.5
Sustaining CAPEX	446.3	439.2		425.0	417.9
All capital costs	488.1	460.3		404.1	376.1

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 H2 2027. Production stoping will ramp up

through 2027, reaching full production in H1 2028. Additional details are provided in Table 1-26 below.

Table 1-26: UG Execution Schedule

Description	2025	2026		2027		2028
	H2	H1	H2	H1	H2	H1
OPEN PIT						
OPEN PIT OPERATIONS						
SHAFTS						
PROD SHAFT - SINKING						
PROD SHAFT - EQUIP and COMMISSION						
PROD SHAFT - SKIP HOISTING						
VENT SHAFT - SINKING						
VENT SHAFT - EQUIP and COMMISSION						
DEVELOPMENT						
670 LEVEL						
470 LEVEL						
405 LEVEL						
310 LEVEL						
285 LEVEL						
INFRASTRUCTURE						
MAIN FANS						
PUMP ROOM						
FINE ORE BIN						
CRUSHER						
CONVEYOR						
WORKSHOP						
UGP PRODUCTION						
DRAWBELLS						
UNDERCUT						
STOPING						
PRODUCTION ACHIEVED (75%)						

1.17 Conclusions

It is the conclusion of the QPs that this 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 2023 FS, the UGP has advanced in terms of detailed engineering, procurement, 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 shaft equipping, underground development, and securing production equipment and workforce with continued focus on meeting and improving the Project schedule and budget.

The most significant potential internal (controllable) risks associated with the Project are; accuracy of the Mineral Resource estimate, hydrogeological, and geomechanical estimates, cave performance, uncontrolled dilution, operating and capital cost escalation, schedule delay, the ability to grout and manage water inflows, 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

The UGP is economically viable under the economic assumptions presented herein 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. Future mining operations are planned in host rock that is relatively unfamiliar to Karowe and it will be important to gather and analyze new data to bolster predictive models and inform planning efforts.

Notable technical recommendations are summarized below. While some recommendations may extend beyond the current scope of this FS, they are nonetheless valuable and should be considered to further enhance the quality of project data that supports operational decision-making.

- Continue to reconcile production forecasts with actual mine output to improve mine plans and revenue forecasts;
- Evaluate further drilling, mapping, and geological assessment to refine resource distributions, model extents, and rock properties at extraction and undercut levels;
- Bulk sample inferred resources during lateral development campaigns to improve resource definition;
- Install groundwater monitoring instruments and flow metres to track depressurization and inflow rates, and continue to monitor water chemistry for maintenance, treatment, and procurement planning;

- Advance geotechnical instrumentation drilling and install monitoring devices to track extraction area and cave performance;
- Advance detailed drill & blast design to inform equipment and consumable supply and contractor scope details;
- Update the Cave Management Plan as new information becomes available and use it to inform the mine planning and operational processes;
- Advance training of mine operational staff to operate and maintain infrastructure as it is commissioned, and to self perform technical services tasks;
- Engage with underground mine construction and production drilling contractors to confirm availability and rate assumptions; and
- Reconcile shaft cycle times, capacities, and availabilities following successful commissioning in 2026 to inform work schedules, mine plans, and production forecasts.

2 INTRODUCTION

JDS was commissioned by Lucara Diamond Corp. (Lucara) to lead an updated Feasibility Study (FS) for the Karowe Diamond Mine (KDM) UG Project (UGP or Project) currently being built to establish UG mining after the completion of OP mining. This Report describes the combined LOM for the OP and UGP as well as highlights the contribution of the UG to the overall project economics.

This Report is updated from the 2023 UGP FS and includes the following notable changes:

- Updated geological model, block model and resource statement;
- Consideration of ongoing Project construction progress (infrastructure and UG development) to the effective date of this Report;
- Changes of the primary Mine Method to increase reliance on natural caving;
- Advancement of detailed engineering work and geotechnical evaluations;
- Re-modelling of the hydrogeological conditions;
- Modifications to the mine, mill and project construction schedules;
- Re-estimation of the current operations budgets and Project capital and operating cost projections; and
- Revised economic modelling with updated diamond prices and exchange rates, exclusion of sunk costs and inclusion of financing costs.

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 September 30, 2025.

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 Qualified Persons (QPs). The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

2.1.1 Scope of Work

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

- JDS Energy & Mining Inc. (JDS):
  - Property Description;
  - Mineral Reserves;
  - Mining Methods;
  - Mine Closure;
  - Mine Infrastructure;
  - Mine Permits;
  - Project Execution;
  - Cost estimation;
  - Economic Modelling; and
  - Report Compilation.
- SRK Consulting (Canada) Inc. (SRK):
  - Geology;
  - Mineral Resource Estimation; and
  - Market Studies.
- Middindi Consulting (Middindi):
  - Geotechnical Analysis.
- Beck Engineering (Beck):
  - Cave modelling.
- Itasca Consulting Group (Itasca):
  - Hydrogeological Modelling.

- K-Met Consultants Inc. (K-Met):
  - Mineral Processing.
- DRA Botswana (Pty) Ltd. (DRA):
  - Recovery Methods.
- Knight Piésold (Pty) Ltd. (Botswana) (KP):
  - Residue Storage Facilities.
- PRIZMA LLC (PRIZMA):
  - Environment and Social Considerations.

2.1.2 Qualifications and Responsibilities

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in NI 43-101, and are members in good standing of appropriate professional institutions / associations. 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)
Matthew Moss, P.Eng.	JDS	Overall Project Management, UG Mining, Infrastructure, Mine Closure, Permits, Project Execution	1-5, 12.10, 12.7, 16 (except 16.3, 16.4), 18 (except 18.4), 20.4, 20.5, 24-27
Brandon Chambers, P.Eng.	JDS	Mineral Reserve Estimate, Capital and Operating Costs, Economic Evaluation	12.3, 12.12, 15, 21-22
Kimberley Webb, P.Geo.	SRK	History, Geology, Deposit Types, Exploration, Drilling, Sample Preparation, Adjacent Properties	6-11, 12.1, 23
Hermanus Grütter, P.Eng.	SRK	Mineral Resource Estimate, Market Studies	12.2, 14, 19
Houmao Liu, Ph.D., PE	Itasca Denver	Hydrogeological Considerations and Water Management	12.6, 16.4
Trevor Rangasamy, MSc, FSANIRE, MSAIMM	Middindi	Geotechnical Analysis	12.5, 16.3

QP	Company	QP Responsibility / Role	Report Section(s)
Kelly McLeod, P.Eng.	K-Met	Mineral Processing	12.4, 13
Lehman van Niekerk, Pr. Eng.	DRA	Recovery Methods	12.9, 17
Justin Teixeira, Pr. Eng.	KP	Residue (Tailings) Storage Facilities	12.11, 18.4
Mehrdad Nazari, MSc, MBA, MIMMM	PRIZMA	Social, Environment	12.8, 20 (except 20.4, 20.5)

2.2 Qualified Person Site Visits

All QPs visited the KDM as per Table 2-2, except for Kelly McLeod, as the comminution systems have been operational for over ten years with no plans to change.

Table 2-2: QP Site Visits

Qualified Person	KDM Visit Dates	Description of Inspection
Kimberley Webb	Jun 14 - 22, 2017 Jun 11 - 15, 2018 May 8 - 17, 2019 Oct 29 - Nov 6, 2024 Apr 22 - 30, 2025 Nov 10 - 20, 2025	• Design kimberlite core logging procedures, geological sampling protocols, and train geologists; • Review OP exposures, mapping procedure, and stockpiles; • Examine kimberlite drill cores from all drilling programs; and • Review Lucara's diamond sales and marketing office in Gaborone.
Hermanus Grütter	May 1 -3, 2024	• Review OP geology mapping, OP survey, OP depletion and truck tally protocols. Review core logging, sampling, record-keeping and related database controls. Tour of Main Treatment Plant. Overview of in-progress shaft sinking.
Brandon Chambers	Aug 28 - Sep 6, 2019 Regular visits between 2022 - 2024	• Full update, review and inspection of the UGP, OP and existing facilities; • Regular meetings and discussions with various technical and management personnel, inclusive of the on -site OP engineering team and site contractors; and • On-site engineering, procurement, and construction management of the UGP.
Matt Moss	Regular visits (<quarterly) between 2020-2024 Most recent Nov 23, 2024	• 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.

Qualified Person	KDM Visit Dates	Description of Inspection
Houmao Liu	Apr 2022	• Visited the OP and dewatering operations; • Visited the tailing facilities; • Examined cores; and • Met with hydrogeologist, mine planning, and Geotech teams at the mine regarding depressurization of pit slope.
Trevor Rangasamy	Feb 24 - 28, 2025 (Regular visits by nominated engineers to geotechnically log and QA/QC outsourced logging)	• Geotechnical inspection of drill core and core logging to derive Kimberlite fracture frequencies, pit wall inspections to assess as built versus design compliance, portal location site inspections, trade-off meeting for portal location, meetings with General Manager, Mine Manager and Project Manager to scope the 2025 geotechnical drilling programme, collection of historical geotechnical database for revision and update.
Kelly McLeod	No mine site visit	• Visited the metallurgical lab during comminution testing.
Lehman van Niekerk	Sep 2-3, 2019 Oct 14-16, 2025	• Review of the surface treatment plant process and discussions with various technical and management personnel.
Justin Teixeira	Dec 12, 2018 Sep 2 - 3, 2019 May 09 - 11, 2023 Jan 29 - 31, 2024 Sep 17 - 19, 2024 Nov 24 - 27 2025	• Project scope, Slimes and tailings operation review, information gathering from various technical/plant personnel; • UGP expansion project scope (FRD3), Slimes and tailings operation review, information gathering from various technical personnel; • Slimes and tailings operation review, construction QA/QC inspection; and • Slimes dams site inspections, construction QA/QC site inspections.
Mehrdad Nazari (MN), supported by Dr. Donald Proebstel (DP)	Apr 27-28, 2021 (MN, DP) Feb 14-25, 2022 (MN, DP) Feb 13-17, 2023 (MN, DP) Nov 21-25, 2024 (DP)	• Engagement with site staff and stakeholders to verify EIA, SIA and EMP findings; • Examination of site conditions; • Verification of MAC TSM reporting and Human Rights Reviews; and • Examination of consultant procedures to generate monitoring data and findings.

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 28.

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, the QP 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 two 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 2024-2025 field programs;
- Review of diamond production and diamond sales data;
- Regional and international vendors;
- Past internal and external reports including the 2023 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.

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 Report. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

As the Karowe Mine has been in active operation for over a decade, the Company has well established processes for managing and updating KDM specific information, costs, and budgets, on which the QPs have relied, including:

- Mineral Licences and fiscal regimes, informing Section 4.1;
- Permitting Rights and Agreements, informing Sections 4.3;
- Project infrastructure, construction history, and detailed engineering completed to date, informing Section 18.1;
- Production and diamond sales history and prices, informing Sections 6.7, 14.5 and 19;
- Operational costs for G&A, OP mining, processing, corporate, and sustaining project capital, informing Sections 21.1 and 21.2; and
- Economic analysis and tax estimation, informing Section 22.

Non-QP specialists relied upon for specific advice are listed below, along with the extent of their involvement and the sections of the report to which their input applies:

- Beck Engineering (Beck) provided geotechnical modelling and cave simulation work summarized in Beck (2025). The information contributes to Sections 15.2.2, 15.3.2, 16.3.8, 16.3.9, and 16.12.3;
- Digby Wells Environmental (Digby Wells) provided closure plans and cost estimates summarized in Digby Wells (2024). The information contributes to Section 20.4 and 21.1.12;
- Aqualogic (Pty) Ltd-Consulting (Aqualogic) provided groundwater flow models summarized in Aqualogic (2025). The information contributes to Section 16.4; and
- United Mining Services (UMS) provided Shaft detailed engineering, cost, and schedule. The information contributes to Sections 16.7.2and 21.1.3.2.

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

4 PROPERTY DESCRIPTION AND LOCATION

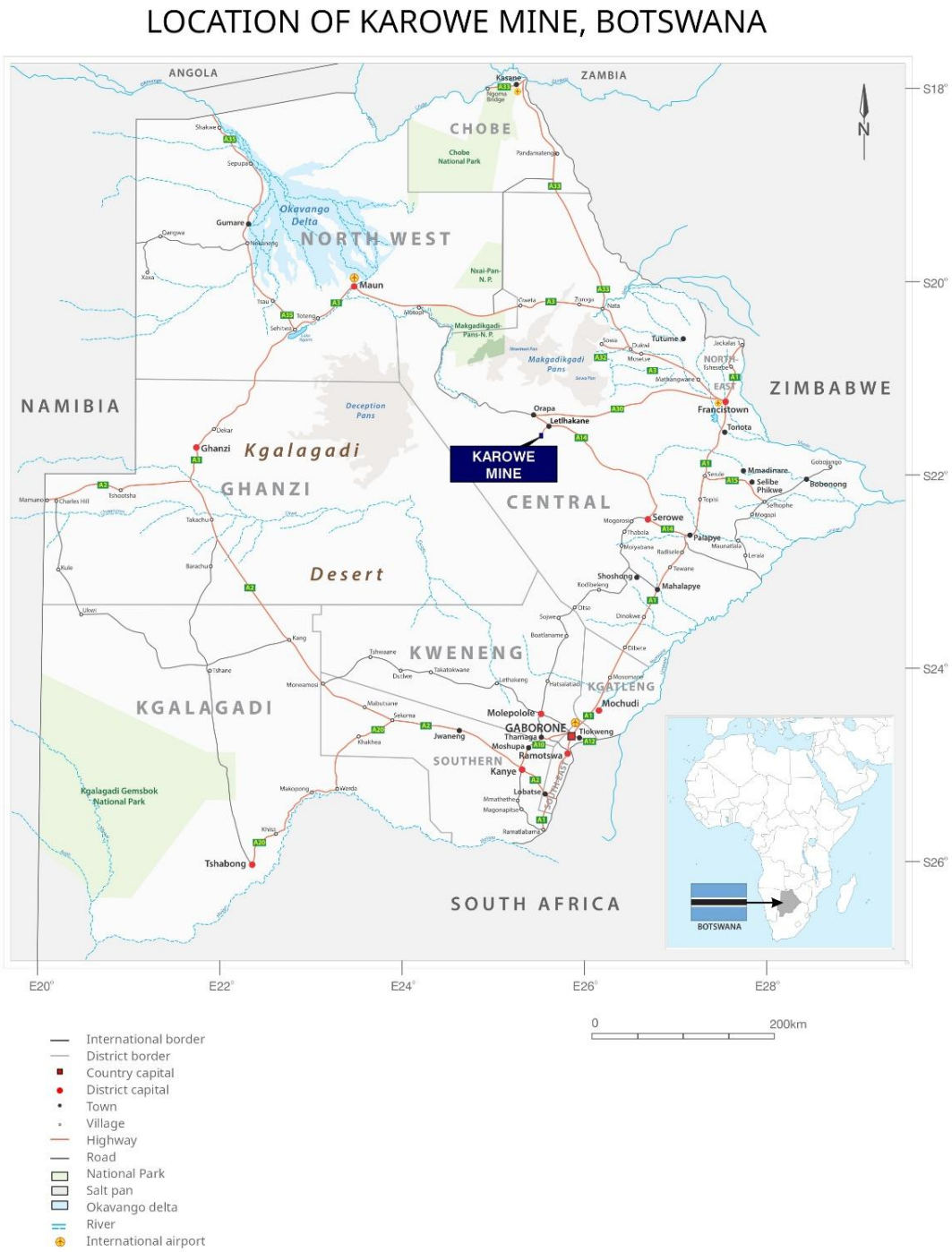
4.1 Location

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 town 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 with the Botswana Democratic Party (BDP) ruling from 1966 to 2024, followed by an elected shift of power to the Umbrella for Democratic Change (UDC) in the most recent 2024 elections. 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 in an equal partnership of the De Beers Group and the Government of the Republic of Botswana. Mining is governed by the *Mines and Minerals Act* and this *Act* is considered one of the most competitive and best administered pieces of mining legislation in Africa. The mining laws are geared to ensure stability, deregulation, government transparency, and were most recently updated in 2024 to enhance economic benefit for Botswana, citizen participation, and penalize defaulting companies.

Figure 4-1: Karowe Diamond Mine Location Map



Source: Lucara (2025)

4.1.1 Types of Mineral Licence in Botswana

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

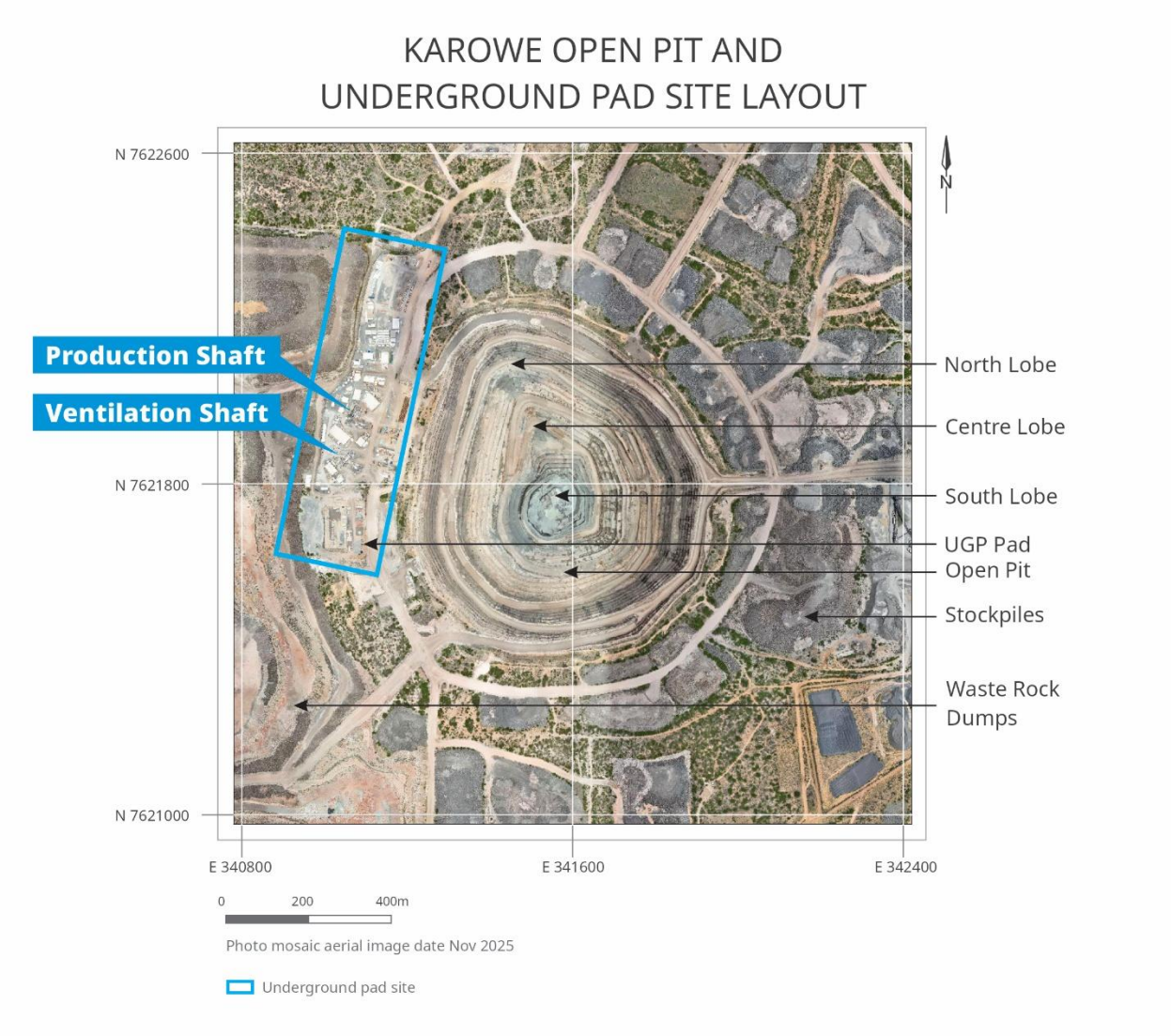
- Prospecting Licence: A prospecting licence is valid for an initial period of up to three years with two renewals each not exceeding two years each. At the end of each period, the prospecting area is reduced by half or at lower proportions as the Minister may decree. The applicant must have access to, or have adequate financial resources, technical competence and experience to carry out an effective exploration program;

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

- Mining Licence: This licence is initially valid for a period of up to 25 years, as is reasonably required to carry out the mining program. The holder of a licence may apply for unlimited reviews for a period up to 25 years. Additionally, mineral rights holders may be required to permit the government to hold up to a 15% minority interest in mining undertakings. This will be on commercial terms with the Government of Botswana 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 (2025)

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 (see Section 22);
- 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 14% 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 Licence

The Property is governed by Mining Licence (ML) 2008/6L, issued in terms of the *Mines and Minerals Act* 1999, Part VI, and covering 15.2 km^2 (1,523 ha).

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

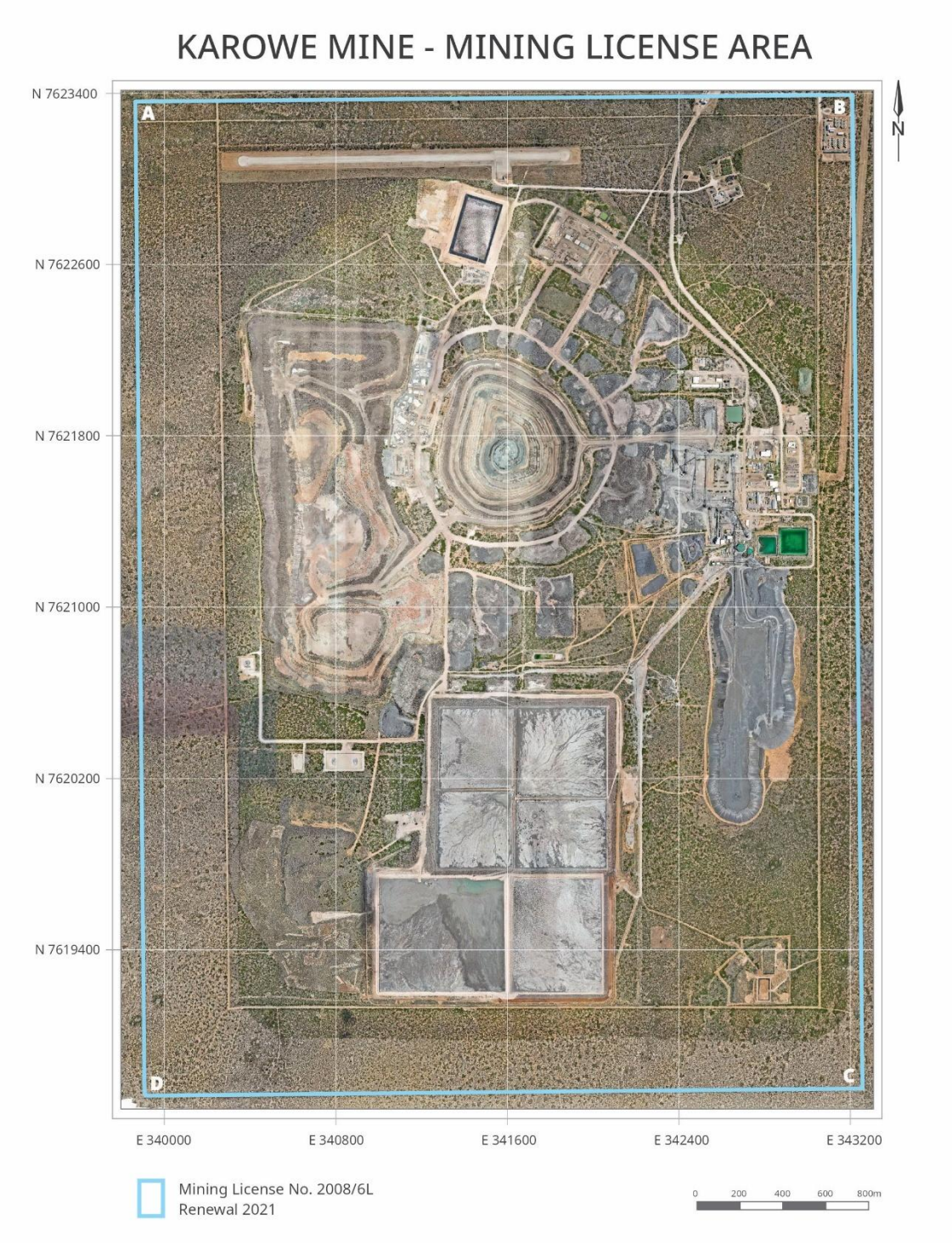
ML 2008/6L is currently 100% held by Lucara Botswana (Pty) Ltd, a company incorporated in Botswana. The ML was originally issued to Boteti Mining (Pty) Ltd for a period of 15 years commencing on October 10, 2008. Boteti Mining was renamed Lucara Botswana on 24 July 2018. The licence was renewed on January 04, 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	CAPE Lo25		UTM WG84/TRS89		Latitude/Longitude (Deg, Min. Sec)	
	Y	X	E	N	LAT	LONG
A	-47,112	2,376,665	339,862	7,623,267	21:29:09.770	25:27:15.260
B	-50,459	2,376,675	343,209	7,623,300	21:29:09.774	25:29:11.544
C	-50,445	2,381,226	343,253	7,618,750	21:31:37.720	25:29:11.544
D	-47,099	2,381,216	339,907	7,618,717	21:31:37.723	25:27:15.260

Source: Lucara (2021)

Figure 4-3: Aerial View of the KDM Mining Licence Area



Source: Lucara (2025)

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. Lucara Botswana has obtained common law land rights for the ML 2008/6L surface area and the access road. These rights will remain in force during the entire mine lease, which is currently in effect 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). KDM 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 licence 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 licence 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 of 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. Many permits are annual in nature and require annual renewals, for which Lucara maintains the following Permit Compliance register.

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 Protection	*EIA Act*
EIS Approval for Mechanical Evaporators	DEA/BOD/CEN/EXT /MNE 103 II (08)	28-Aug-29	Dept. of Environmental Protection	*EIA Act*
Water Rights	B6316, B6317, B6023, B6615, B6622, B5386, B 5387, B5388, B5389, B7942, B7933, B7934, B7935, B7936, B7937, B7938, B7939, B7940, B7941, B7942	Valid for the duration of the mining licence	Dept. of Water Affairs	*Water Act*
Borehole Certificates	Z18429, Z18430, Z18431, Z18432, Z18433, Z18434, Z18435, Z18436, Z18437, Z18438	Valid for the duration of the mining licence	Dept. of Water Affairs	*Boreholes Act*
Dumps Classification	All classified	All dumps active	Dept. of Mines	*Mines, Quarries, Works and Machinery Act*
Surface Rights	LT/SLB/B/1 IV (231)	ML years	Ngwato Land Board	*Tribal Land Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Incinerator Permit	DEP 7/1/32 II	24-Jun-27	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Waste Water Treatment Plant	WMF01/2022/11/20-WWTW/Karowe Diamond Mine	24-Dec-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Landfill	WMD/22-2024/304-10/LF/Letlhakane	31-Dec-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Salvage yard	WMF/20-2024/20-12//Letlhakane	31-Dec-25	Department of Waste Management Pollution Control	*Waste Management Act*
Permit to purchase, acquire and Posess Explosives	F001/2022 E-PPAP0594/2024	31-Dec-25	Dept. of Mines	*Explosives Act*
Permit to carry bulk explosives	E-PCE0161/2025 Vehicle No: B868BOY	30-Dec-25	Dept. of Mines	*Explosives Act*
Explosives magazine licence	E-MLF0047/2025	31-Dec-25	DME	*Explosives Act*
Explosives magazine licence	E-MLF0046/2025	31-Dec-25	Dept. of Mines	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	E-MLF0047/2025	31-Dec-25	Dept. of Mines	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	00003647A	31-Dec-25	DME	*Explosives Act*
Permit to import and possess explosives	E-PIPE0584	31-Dec-25	DME	*Explosives Act*
Application for restricted blasting licence		N/A	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B681BMU E-PCE0163/2025	30-Dec-25	DOM	*Explosives Act*
Permit to carry explosives in Bulk	E-LMO0033/2025 Vehicle No: B339BPM	31-Dec-25	DOM	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B429BJB E-PCE0162/2025	31-Dec-25	DOM	*Explosives Act*
Licence to manufacture explosives	E-LMO0034/2025 Vehicle No: B693BRO	31-Dec-25	DME	*Explosives Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Box storage for conveyance and Storage of explosives	F01/22 F02/22 F03/22 F04/22	As and when required	Dept. of Mines	*Explosives Act*
Blasting Licence for magazine master	In Place	Valid and appointment renewed annually	Dept. of Mines	*Explosives Act*
Airstrip Licence	B509	Jun-26	Civil Aviation	*Aviation Act*
Generator Licences	CBERA 7/2/93 I	Once off	BERA	*BERA Act*
Solar photovoltaic plant	CBERA 7/2/89 I	Once off	BERA	*BERA Act*
Standby Generator Licences	CBERA 7/2/72 I	Once off	BERA	*BERA Act*
Mining Licence	2008/L6	January-46	Dept. of Mines	*Mines & Minerals Act*
Licence to possess and use radioactive sources	BW0611/2024	31-Jul-26	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 44:04
Kibble Winder 10 - 069 - Production shaft	M35 M 1	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Stage Winder 10 - 071- Ventilation shaft	M35 M 1 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Stage Winder 10 - 087/88 Ventilation shaft	M35M I	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Approval letter for charging units	11-May-22	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*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
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 XXVII (55)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Chemical clearance letter	HSM 6/6/25 V (1)	N/A	Department of Public Health	*Public Health Act*
Kibble Winder 10 - 069 - Production shaft	DOM 6/13/5 I(13)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Kibble Winder 10 - 0399 – Ventilation shaft	DOM 6/13/5 I(12)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Permit to purchase, acquire and Posess Explosives	E - PPAP0035/2024	31-Dec-25	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	E-PCE0161/2024	31-Dec-25	Dept. of Mines	*Explosives Act*
Kibble Winder 10 - 069 Production shaft	DOM-6/13/ 51(13)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Approval of Diesel Engine Unit use in Underground Sinking Operation	DOM - 6/13/ 51 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Permit to export/import/transit controlled ozone depleting substances	00000696A	31-Dec-25	Metrological Services	*National Metrological Services Act* of 2014
Chemical clearance letter	HSM 6/6/15 V (1)	31-Dec-25	Department of Public Health	*Public Health Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Remotely Piloted Aircraft Certificate	RPA (H)950 RPA (H)958 RPA (H)957 RPA (H)951	28-Feb-26	CAAB	*Civil Aviation Act*
Remotely Piloted Aircraft Certificate	RPA (H)953 RPA (H)968	31-Mar-26	CAAB	*Civil Aviation Act*
General Nurse Practicing Certificate	GN. 11621 GN. 4646	31-Dec-25	Nursing and Midwifery Council of Botswana	*Nurses and Midwives Act*

Source: Lucara (2025)

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

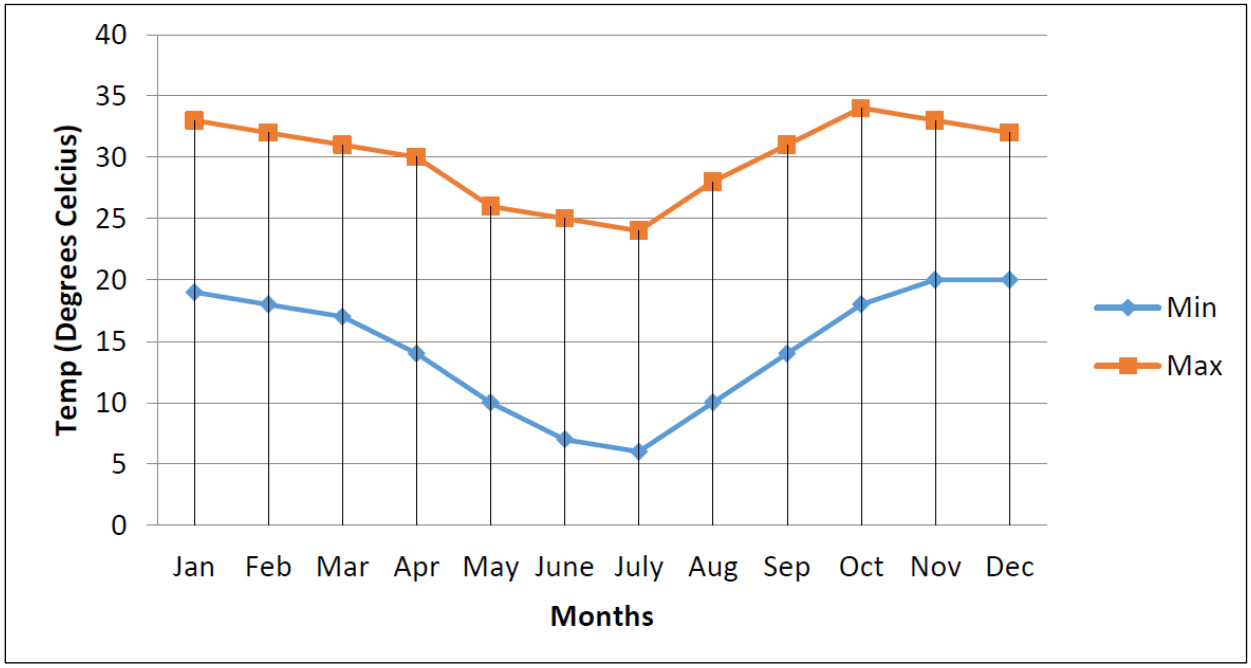
The climate in the Project area is hot and semi-arid, with an average annual rainfall of about 400 mm falling between November and March. Rainfall often occurs in short-duration, intense downpours from discreet cells often associated with lightning and is heaviest in January, February and March.

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 Figure 5-1.

Figure 5-1: Letlhakane Monthly Temperature Averages



Source: Lucara (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, boomslang, 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 2022, the census noted that Letlhakane had a population of 36,400, which is an increase of 36% from the previous census in 2011 prior to KDM entering operations.

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 Licence 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 this section 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 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 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 delineation of kimberlites discovered previously, as well as 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).

One vertical core hole 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 licence 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 licence 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).

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 licence 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 licence 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 licences 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 0 through 11.

A Mining Licence 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 Licence over the AK6 kimberlite. On September 9, 2008, the Government informed Boteti that it would regard the period since the Retention Licence application as a negotiation period as allowed under Section 50 of the *Act* and urged Boteti to apply for a Mining Licence. This was done, and 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 US$49 million. 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 US$7.3 million. In addition, African Diamonds acquired Wati Ventures and its interest of 1.351% to bring their total shareholding in Boteti up to 40%.

In November 2010, Lucara and African Diamonds approved a plan for the construction of the Karowe Mine with full commissioning targeted for early 2012. On 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 for a sum of US$130 million (Lucara, 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 US$4.5 million, 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 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 for overall process improvement and recovery of large diamonds. In November of 2015, Karowe Mine recovered the 1,109 ct gem quality Lesedi la Rona (sold for US$53 million) and the 813 ct Constellation diamond (sold for US$63 million).

During 2017, a drilling program was initiated 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 update of Inferred resources to Indicated resources between depths of 400 to 600 m below surface (600 to 400 masl), including a significant increase in the volume of the higher-grade EM/PK(S) unit (MSC, 2018; Webb et al., 2024). 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 maximise 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 at the time.

In November 2019, Lucara announced positive results from a FS for an underground operation at Karowe Mine – the 2019 Karowe Underground Project FS (2019 UGP FS).

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 an 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 weight. Under this agreement, +10.8 ct stone production from the Karowe Mine is 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 named “Sethunya” of exceptional purity and an unbroken 998 ct diamond, both from direct milling of South Lobe EM/PK(S) ore.

Work on the Karowe underground expansion project continued with an investment of US$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 Licence No 2008/6L in respect of the Karowe Mine had 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, 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 were 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 US$220 million 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 support on-going operations.

Two equity financings were closed in July 2021 that generated net proceeds of US$31.3 million 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 the US$220 million senior secured project debt financing.

In February 2024, Lucara announced it had entered into a 10-year new diamond sales agreement in respect to all qualifying +10.8 ct diamonds.

In March 2024, Lucara announced an updated FS for the Karowe UGP with an effective date of June 30, 2023 (Doerksen et al.,2024). The 2023 FS incorporated project progress to that date, re-baselined the project schedule and budget, and updated the economic modelling.

Open pit operations have continued and are scheduled to end in 2026.

Since the onset of commercial production to the end of Q3 2025, the Karowe Mine has produced 5.08 Mct from 34.8 Mt of processed kimberlite and has sold via tender a total of 4.87 Mct for a total revenue of US$2.65 billion, resulting in an achieved sold average price of US$545/ct (Table 6-1). Significant diamond recoveries are summarised in Section 6.7.1 below.

6.7.1 Significant Stone Recovery to September 30, 2025

From inception to the end of Q3 2025, the Karowe Mine has recovered 43 diamonds > 300 ct, 69 diamonds between 200 and 300 ct, and 283 diamonds between 100 and 200 ct. The mine has recovered nine diamonds in excess of 1,000 ct since 2015.

Since 2012, the mine has produced over 290,000 ct and over 9,000 stones in excess of 10.8 ct for an average stone size of 30.9 ct/stn for the +10.8 ct production. In the period from 2015 to September 30, 2025 inclusive, the mine has averaged 6.4 wt% of total production being >10.8 ct in weight, and 7.5 wt% of M/PK(S) and EM/PK(S) production. The nine stones exceeding 1,000 ct include a 1,174 ct diamond recovered in 2021, a 1,080 ct diamond recovered in 2023, and the 2,488 ct Motswedi and 1,094 ct Seriti recovered in Q3 2024.

Table 6-1: Karowe Mine Production and Sales Results

Year	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025*	Total
Kimberlite Mined (tonnes)	1,600,971	3,944,343	3,327,754	3,187,222	2,722,375	1,575,052	3,113,362	3,303,375	2,987,775	3,668,677	3,308,254	2,700,325	3,001,727	1,628,805	40,070,017
Waste Mined (tonnes)	4,074,196	5,493,445	10,270,720	13,890,115	11,058,041	15,865,121	15,001,820	6,542,781	2,662,327	2,619,744	1,493,113	3,079,452	944,082	96,191	93,091,148
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	2,849,135	2,850,631	2,082,731	34,798,523
Carats Recovered	294,167	440,751	430,292	365,690	353,974	249,767	366,086	433,060	381,707	369,390	335,768	395,134	389,017	276,391	5,081,194
Recovered Grade (cpht)	22	19	18	16	14	11	14	15	14	13	12	14	14	13	15
Carats Sold	152,724	438,717	412,136	377,136	358,806	260,526	350,799	411,736	372,941	380,493	327,027	379,287	399,215	251,460	4,873,003
Sales Average US$/ct**	$274	$411	$644	$594	$823	$847	$502	$465	$334	$599	$623	$455	$511	$498	$545
Total Revenue US$***	$41,830,289	$180,506,581	$265,504,005	$223,846,419	$295,465,574	$220,762,844	$176,191,233	$191,603,268	$124,492,536	$227,977,391	$203,803,042	$172,399,847	$203,876,389	$125,208,535	$2,653,467,953

Notes:
* To end September 2025.
** Sales average US$/ct represents a rounded figure. Total revenue will not recalculate from this amount.
*** Total revenue reflects revenue from Karowe goods only and does not include revenue from the Clara sales platform from the date of acquisition to its disposal in October 2024.

Source: Lucara (2025)

7 GEOLOGICAL SETTING AND MINERALIZATION

This section is restated or summarised from the previous Technical Report (Doerksen et al., 2024) with the addition of an updated structural model, updated descriptions of kimberlite units based on recent drilling and pit mapping, updates to the South Lobe geological model since the 2019 FS (Doerksen et al. 2019) and an advanced emplacement model.

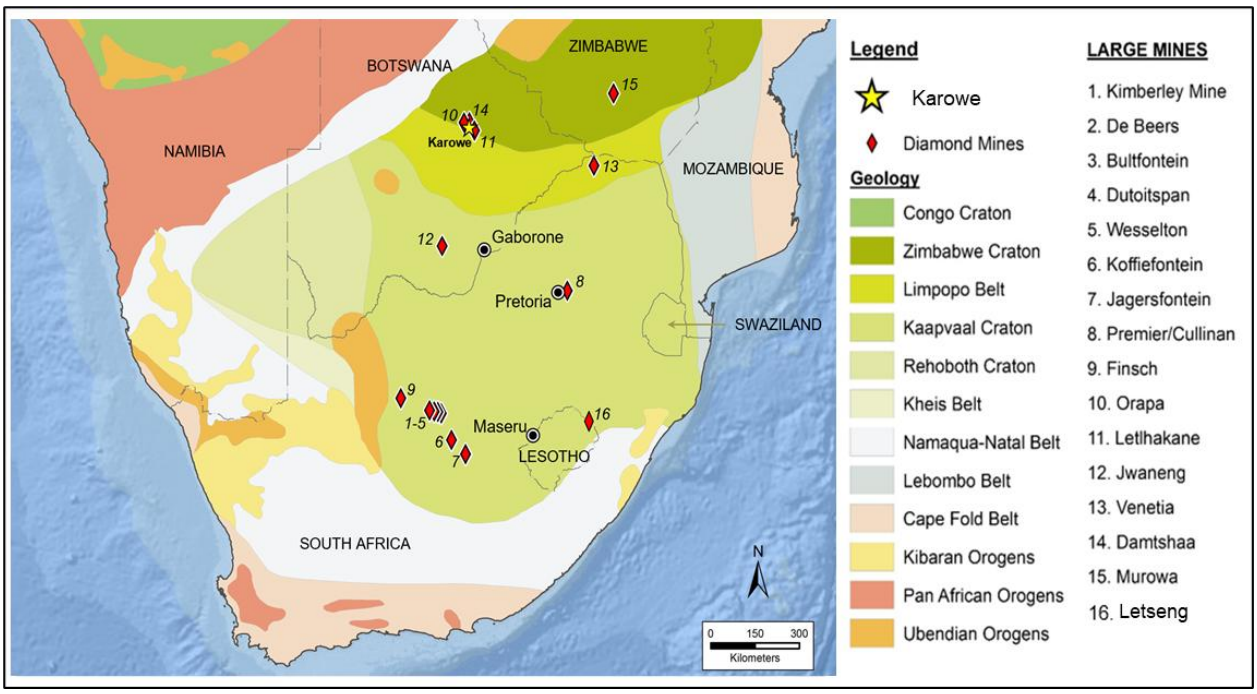
7.1 Regional Geology

KDM is exploiting the AK6 kimberlite, which is part of the mid-Cretaceous Orapa Kimberlite Field (OKF) in the Central District of Botswana on the western edge of the Zimbabwe Craton (Figure 7-1). The OKF comprises 86 known kimberlite bodies of post-Karoo age that occur over an area of approximately 2,600 km^2 along the southern shore of the Makgadikgadi Basin. The kimberlites occur in several clusters and include bodies of varying size (more than 100 ha to less than 0.5 ha), mode of emplacement and diamondiferous character. Nine of the kimberlites have been or are currently being mined – AK1 (Orapa), AK6 (Karowe), BK1, BK9, BK12, BK15 (Damtshaa), DK1 and DK2 (Letlhakane), and BK11. The OKF lies on the northern edge of the Central Kalahari Karoo Basin, where the Karoo succession dips gently to the SSW and off-laps against Precambrian basement rocks that occur at shallow depth but are rarely exposed within the Makgadikgadi Depression. The condensed Karoo succession has a total thickness of ~600 m and is best preserved in WNW-ESE oriented grabens such as that hosting the Orapa Mine 25 km northwest of Karowe (Coates et al., 1979).

The regional stratigraphy is shown in Table 7-1. The flood basalt of the Stormberg Lava Group is underlain by 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 Karoo stratigraphy reflects the transition from glacial conditions to fluvio-deltaic, swampy, and ultimately arid aeolian environments. 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 deformation history of Central Botswana, leading up to the emplacement of the kimberlites in the mid-Cretaceous, and post-kimberlite deformation with ongoing neotectonics, provides the structural framework and understanding of structures at Karowe.

Figure 7-1: Location of Karowe on the Western Edge of the Zimbabwe Craton



Notes:
The figure shows the major tectonic provinces of southern Africa (Artemieva and Vinnik, 2016).
Source: SRK (2025)

Major lithospheric structures, such as the NE-trending Southern Trans-Africa Shear System (STASS), have been repeatedly reactivated since the Permian, influencing basin development, deposition and kimberlite emplacement. During the Cretaceous, continental break-up produced transform faults and grabens across southern Africa. During this time, the STASS was again reactivated, preserving Karoo basin rocks in NE-SW trending grabens and together with other lithospheric structures controlled kimberlitic and related alkaline magmatism (Moore et al., 2008, Jelsma et al., 2009) and uplift and erosion. Neotectonic activity, driven by intra-continental rifts and mid-ocean ridges, continues to shape the region, with NW-SE trending faults dominating the structural framework.

Table 7-1: Regional Stratigraphy of the Central District of Botswana

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

Source: McGeorge, et al. (2010)

7.2 Property Geology

7.2.1 Country Rock Geology

Drilling has defined the country rock succession at the KDM property as shown in Table 7-2 and Figure 7-2. The volcanic and sedimentary units are almost flat lying. The Stormberg basalts dominate the upper stratigraphy, while the underlying Karoo sedimentary rocks include the Ntane Sandstone, Mosolotsane, Tlhabala, and Tlapana Formations, which have a combined thickness of approximately 350 m and are significant at Karowe for their geotechnical and hydrogeological properties. The country rocks are summarised as follows:

- Kalahari Group: Surface sands and calcrete up to 15 m thick;
- Stormberg Lava Group: Basalt flows (up to 125 m thick) characterised by vesicular flow tops, columnar jointing, and weathering zones extending 10–20 m vertically;
- Ntane Formation: Aeolian quartz arenite (~60 m thick) forming a regional aquifer with preserved palaeodunes and cross-bedded strata;
- Mosolotsane Formation: Interbedded mudstones and sandstones (~55 m thick) with swelling clays and weak zones impacting pit stability;
- Tlhabala Formation: Non-carbonaceous grey-green mudstones (~105 m thick) with swelling characteristics, correlated with the lacustrine Beaufort Group;
- Tlapana Formation: Carbonaceous mudstones and coal seams (~130 m thick), representing weak zones and geotechnical challenges;
- Mea Formation: Porous arkosic sandstones (~2 m thick) with thin coal stringers; and
- Basement: Fractured Archaean granite-gneiss (from ~490 m depth below surface) hosting groundwater aquifers and shear zones, with porous leached granite zones occurring discontinuously adjacent to the kimberlite pipes. Leaching is a natural chemical weathering process commonly observed around kimberlite intrusions as a result of reaction between volatile-rich fluids released from the magma and circulating groundwater with the granite host rocks.

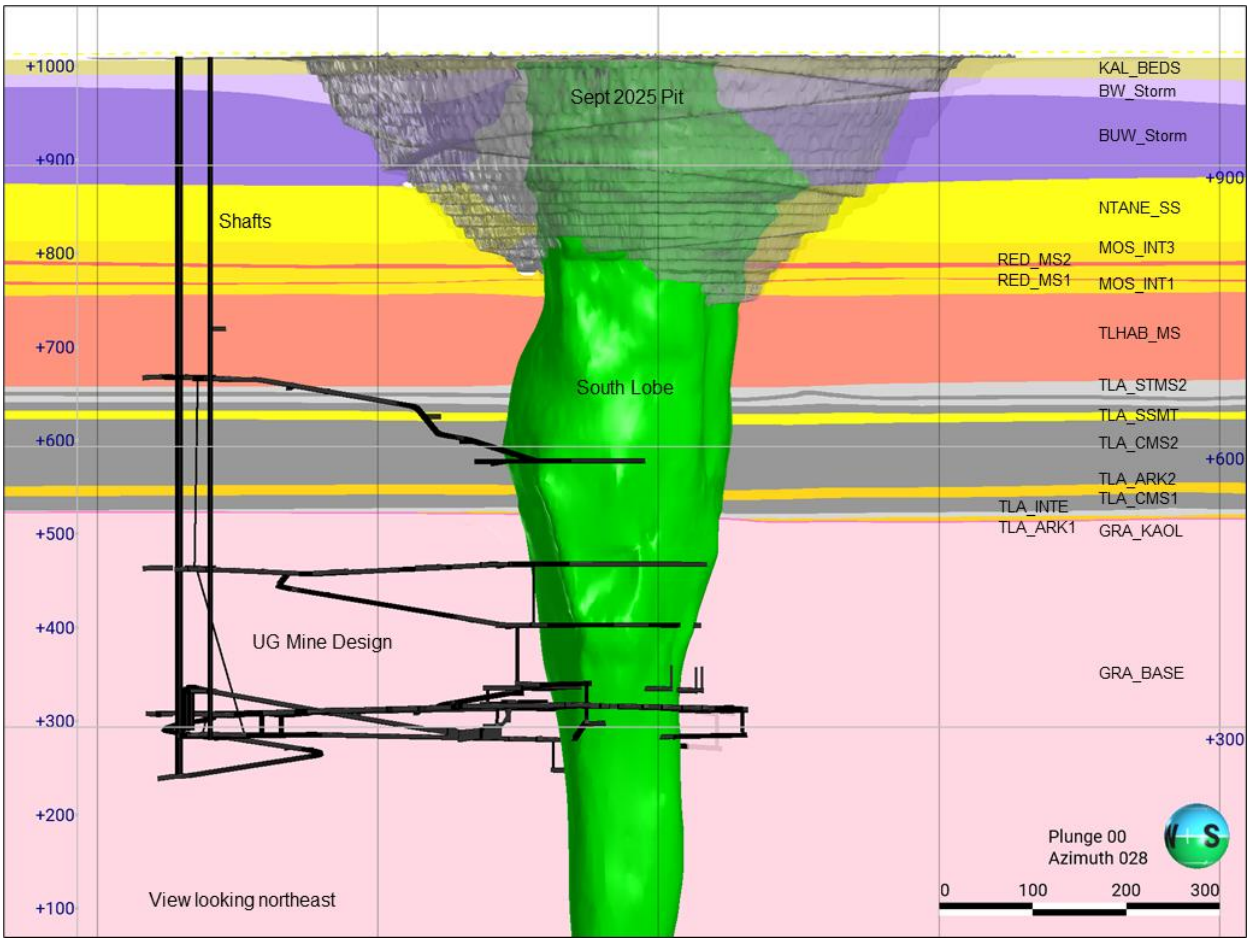
Table 7-2: Stratigraphic Thicknesses at the KDM Property

Stratigraphic Unit	Depth from Surface (m)	Elevation (masl)	Thickness (m)
Kalahari Group	8 – 15	1015 - 1000	8 – 15
Stormberg Lava Group (Basalt)	15 – 135	1000 - 880	120
Lebung Group (Ntane Sandstone and Mosolotsane)	135 – 255	880 - 760	120

Stratigraphic Unit	Depth from Surface (m)	Elevation (masl)	Thickness (m)
Tlhabala Formation	255 – 360	760 - 655	105
Tlapana Formation	360 – 490	655 - 525	130
Granitic Basement	>490	> 525	

Source: SRK (2025)

Figure 7-2: Country Rock Geology Model



Source: SRK (2025)

The country rock lithologies and corresponding abbreviations used in this Report are listed in Table 7-3.

Table 7-3: Country Rock Lithologies and Rock Codes

Rock Code	Formation	Rock Types
KAL_BEDS	Kalahari Formation	Kalahari Beds
BW_Storm	Stormberg Group	Weathered Basalt
BUW_Storm	Stormberg Group	Unweathered Basalt
NTANE_SS	Ntane Sandstone Formation	Sandstone/Siltstone
MOS_INT3	Mosolotsane Formation	Sandstone
RED_MS2	Mosolotsane Formation	Red Mudstone
MOS_INT2	Mosolotsane Formation	Siltstone
RED_MS1	Mosolotsane Formation	Red Mudstone
MOS_INT1	Mosolotsane Formation	Mudstone
TLHAB_MS	Tlhabala Formation	Mudstone
TLASTMS2	Tlapana Formation	Sandstone/Mudstone
TLA_CMS4	Tlapana Formation	Carbonaceous Mudstone
TLASTMS1	Tlapana Formation	Sandstone/Mudstone
TLA_CMS3	Tlapana Formation	Carbonaceous Mudstone
TLASS_ST	Tlapana Formation	Sandstone/Siltstone
TLA_CMS2	Tlapana Formation	Carbonaceous Mudstone
TLA_ARK2	Tlapana Formation	Arkose Sandstone
TLA_CMS1	Tlapana Formation	Carbonaceous Mudstone
TLA_INTE	Tlapana Formation	Sandstone/Siltstone/Mudstone/Carbonaceous Mudstone
TLA_ARK1	Tlapana Formation	Arkose Sandstone
GRA_KAOL	Basement (Kaolinitized)	Kaolinitized Granite-gneiss
GRA_BASE	Basement	Granite-gneiss
LGRAN	Basement	Leached Granite-gneiss

Source: Middindi (2025)

7.2.2 Structural Geology

The KDM structural geology model was updated in 2024, employing a hierarchical approach to better capture the natural continuity of geological structures (SRK, 2024). The structural model incorporates previous pit mapping and drillhole structural data, pit wall scans, drone imagery, and new pit and underground mapping data collected during 2024 (Figure 7-3).

Faults were modelled based on direct structural data and observed continuity in the pit walls and scans. A total of 149 structures, including 65 faults, were modelled and six structural domains were identified, reflecting variations in structural intensity, continuity, and lithological context.

Domain boundaries were developed using major structures mapped in the pit and guided by quick joint mapping that involved measuring main joint orientations, persistence and intensity.

The purpose of the updated model was to provide a robust foundation for pit steepening initiatives, risk assessments, and underground design at Karowe by identifying key structures and structural domains influencing pit stability and underground development.

The key features are summarised as follows:

- The structural domain encompassing the central axis of the kimberlite pipes is characterised by a zone of pervasive NNW trending continuous faults and joints (Figure 7-4), as well as a possible ring fault system comprising moderate to shallower small structures. Most of the structures within this zone, known as the 'fracture corridor', terminate at the pipe boundary. The pipe geometry at deeper levels appears to also have been controlled by this trend;

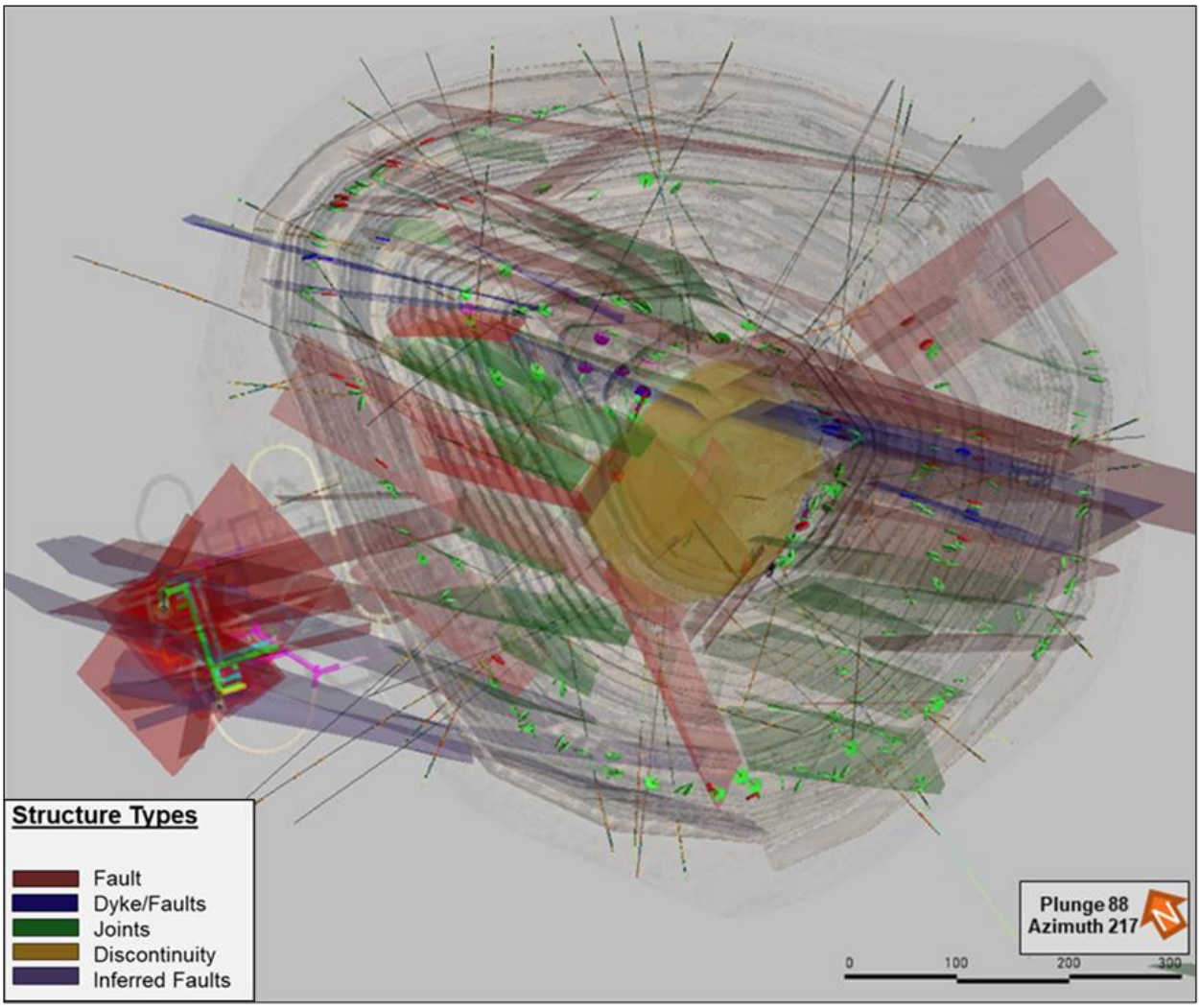
- Steeply dipping NW to NNW strike-slip faults are the dominant structures at Karowe and are inferred to have played a role in kimberlite emplacement. These faults exhibit dextral kinematics and are associated with kimberlite dykes aligned with the NNW structural framework (Figure 7-5);

- Steeply dipping EW to NE trending strike-slip faults are concentrated near the kimberlite pipes and are interpreted as post-emplacement features with limited continuity based on variable slickenside orientation, proximity to the kimberlite pipes and presence of similar structures within the kimberlite;

- Post-emplacement fault displacement is limited within or near the pipes and is mainly associated with the NE-trending structures. Fracture development is likely to have occurred over time during pipe development and during late dyke and fluid emplacement and cooling processes; and

- Low-angle discontinuities within the South Lobe are interpreted as relaxation features along pre-existing lava flow contacts (Figure 7-6).

Figure 7-3: Karowe Structural Geology Model



Note:
Plan view of updated structural model showing interpreted faults and other structures, including larger joints in the pit and near underground infrastructure (shafts; left) with the open pit surface for reference. Structural domains are not shown.
Source: SRK (2024)

Key aspects of the fault model include:

- Modelling Approach: The modelling approach focused on accurately representing the discontinuous nature of faults observed in the pit, rather than exaggerating their continuity based on geophysical lineaments. Faults were modelled to confine smaller structures to areas where they were observed. This approach allowed for greater control over fault continuity, reflecting their segmented nature;

- Fault Orientation: Fault orientations were primarily constrained by field mapping data, remote mapping traces and drillhole structural logging. Where oriented data were unavailable, non-oriented indicators were aligned in 3-D space to construct fault wireframes, though these interpretations were assigned lower confidence levels;

- Fault Continuity: Faults were terminated where mapping or drilling data indicated limited continuity. In areas with uncertain data, observed fault patterns were relied upon. Structures extended to intersect underground infrastructure were identified as inferred trends rather than confirmed fault continuity. Faults were generally terminated at the kimberlite pipe margins; however, where closely spaced observations supported continuity, faults were modelled as observed. This behaviour is more commonly expressed within dyke and joint networks in the structural model;

- Physical Characteristics: Faults were modelled as single planes, though in reality, they often manifest as zones with varying thickness. Most faults in the deposit are narrow, immature, and discontinuous, with thin fault rocks or no fault rocks along their surfaces; and

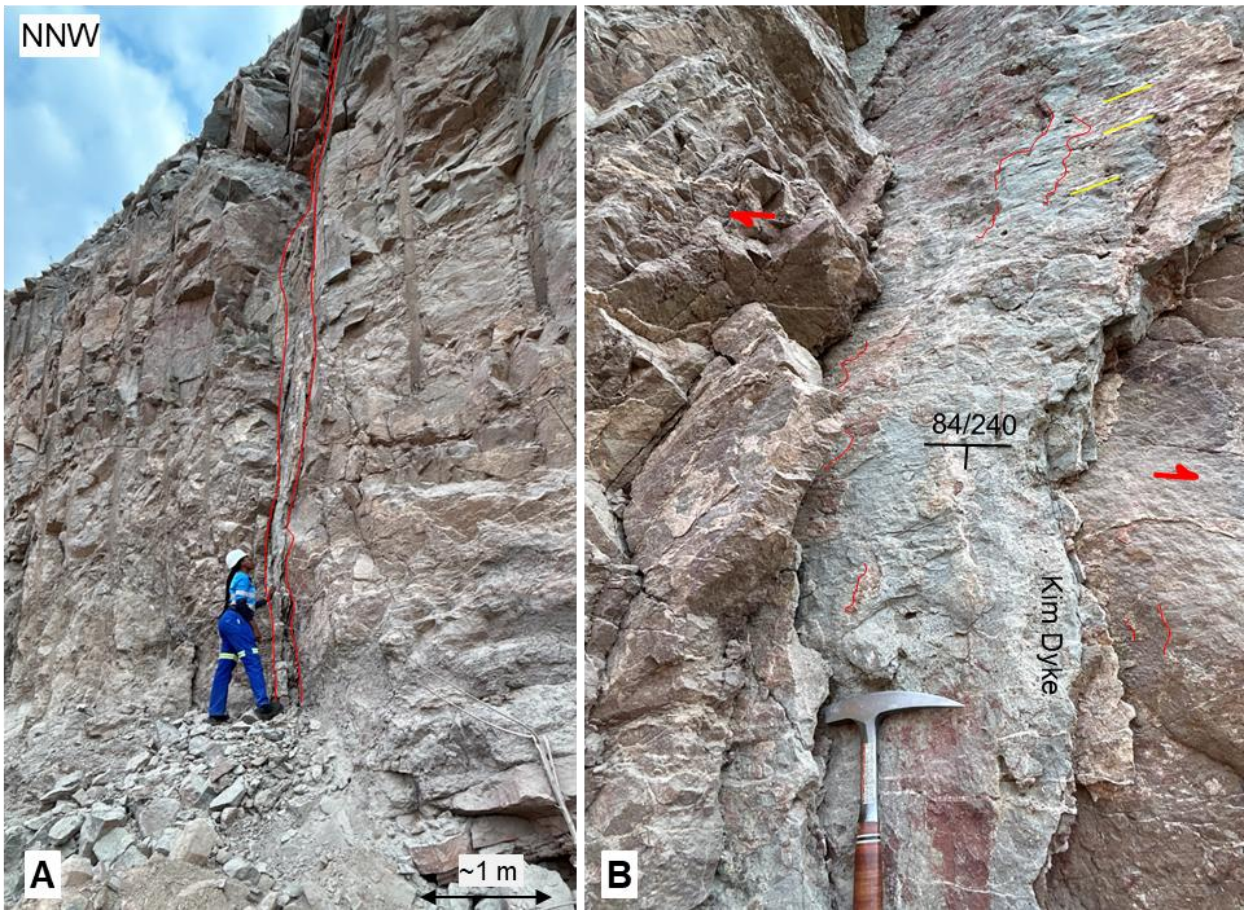
- Data Integration: The model integrates aeromagnetic geophysical data, hydrogeological intervals and regional tectonic and paleostress interpretations to enhance the structural understanding.

Figure 7-4: NNW Faults and Joints in the Fracture Corridor at KDM



Notes:
Viewed from the South Lobe looking north with the North Lobe in the high wall
Source: SRK (2024)

Figure 7-5: NNW Trending Fault and Kimberlite Dyke in Basalt Country Rock



Notes:

NNW trending strike slip fault in the base of the Stormberg basalt with intruded kimberlite dyke that has been subsequently reactivated. A) North pit area with 20 cm kimberlite dyke and later calcite infills, B) Fault kinematic indicators. Note the fault surface steps (small red lines) and slickenlines (yellow lines) that suggest dextral shear sense.

Source: SRK (2024)

Figure 7-6: Low Angle Discontinuities in the South Lobe



Notes:

Overview of the well-developed low angle discontinuities inside the South Lobe interpreted to be a result of relaxation along the pre-existing kimberlite lava flow contacts and some steeper inter-connecting joints. KM = kimberlite; MOS = Mosolotsane Formation

Source: SRK (2024)

7.3 Kimberlite Geology

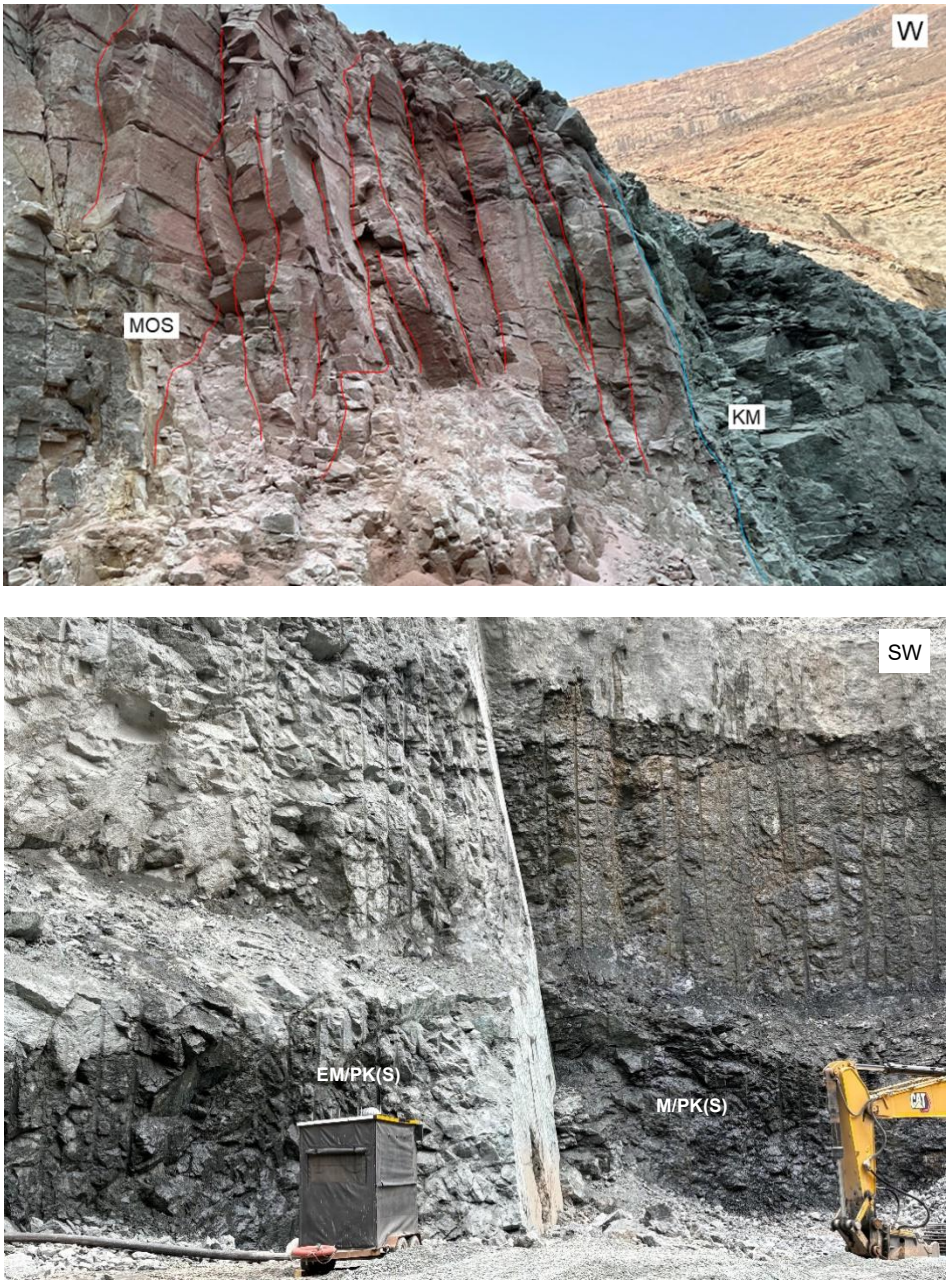
AK6 is a primary diamond deposit that comprises three adjacent, geologically distinct, coalesced kimberlite pipes known as the South, Centre and North Lobes (Figure 7-7). The pipes trend roughly north-south with a combined undepleted surface expression of ~3.3 ha and maximum area of ~8 ha at 120 m below surface. 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 constitutes the current in situ resource. Geological relationships indicate that the North Lobe was emplaced first, followed by the Centre Lobe and South Lobe. The pipe contacts are sharp and distinct, and there is minimal or no brecciation and minimal alteration of the country rock at the contacts (Figure 7-8). AK6 is geologically distinct from the nearby AK1 and other major Orapa field kimberlites in terms of its kimberlite texture despite their comparable age (~90 Ma) and geological setting.

Figure 7-7: View of AK6 in the Open Pit



Notes:
View of South, Centre and North Lobes in the open pit, looking northeast, March 2022
Source: Lucara (2022)

Figure 7-8: South Lobe Pipe Contacts and Internal Domain Contact



Notes:
Top: South Lobe kimberlite contact with Karoo country rock (MOS = Mosolotsane Formation) with a thick zone of fracturing and some brecciation; Bottom: steep contact between the main pipe filling units M/PK(S) and EM/PK(S) in the South Lobe (right and left, respectively)
Source: SRK (2025)

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 more homogeneous than the North and Centre Lobes which exhibit greater textural complexity and more variable and higher proportions of internal country rock dilution. Internal contacts between major units in the South Lobe are steep (Figure 7-8).

The kimberlite in each lobe has been grouped into mappable units (Table 7-4) based on its geological characteristics and interpreted grade potential, including separation of material with very high country rock xenolith dilution (historically termed breccias). Unit identification is based primarily on extensive drill core logging and core photo review, supported by petrographic studies of representative samples (MSC, 2018b; SRK, 2019, 2025), as well as historical groundmass spinel composition analysis and whole-rock geochemical analysis (Hanekom et al., 2006; Tait and Maccelari, 2008). The main geological features of each unit are summarised below. Unless otherwise stated, the kimberlite terminology and olivine and country rock xenolith size and abundance descriptors used are those of Scott Smith et al. (2013, 2018). 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).

The upper weathered and basalt-rich zones in each lobe have been mined out (Table 7-4) and mining has ceased in the Centre and North Lobes, with the remaining in situ kimberlite being excluded from the updated MRE as Unclassified material (Section 14).

Table 7-4 Kimberlite Units Identified in the AK6 Kimberlite

Lobe	Unit	Domain	Description	Status at September 2025
South	BBX	BBX(S)	Country rock breccia	Mined out
	CBBX	CBBX(S)	Calcretized country rock breccia	Mined out
	CKIMB	CKIMB(S)	Calcretized kimberlite	Mined out
	INTSWBAS	INTSWBAS(S)	Large internal block of basalt	Mined out
	WBBX	WBBX(S)	Weathered country rock breccia	Mined out
	WK	WK(S)	Weathered kimberlite	Mined out
	M/PK(S)	M/PK(S)	Magmatic/pyroclastic kimberlite	
	EM/PK(S)	M/PK(S)	Eastern magmatic/pyroclastic kimberlite	
	KIMB3	KIMB3	Minor hypabyssal kimberlite; volume increases below 500 masl	
	WM/PK(S)*	WM/PK(S)	Western magmatic/pyroclastic kimberlite	
	KIMB1*	KIMB1	Volumetrically minor hypabyssal kimberlite	
	KIMB4a*	KIMB4a	Localised variant of EM/PK(S)	
	KIMB5*	KIMB5	Volumetrically minor hypabyssal kimberlite	
	KIMB6*	KIMB6	Volumetrically minor hypabyssal kimberlite	

Lobe	Unit	Domain	Description	Status at September 2025
	RFW-ALT*	RFW-ALT	Highly altered kimberlite of unknown type	
	KIMB7	n/a	Volumetrically minor kimberlite	
Centre	BBX	BBX(C)	Country rock breccia	Mined out
	CKIMB	CKIMB(C)	Calcretized kimberlite	Mined out
	KBBX	KBBX(C)	Kimberlite and country rock breccia	Mined out
	WBBX	WBBX(C)	Weathered country rock breccia	Mined out
	WK	WK(C)	Weathered kimberlite	Mined out
	CFK(C)	CFK(C)	Carbonate-rich fragmental kimberlite	Mined out
	FK(C)	FK(C)	Fragmental kimberlite	
North	BBX	BBX(N)	Country rock breccia	Partly mined out
	CKIMB	CKIMB(N)	Calcretized kimberlite	Mined out
	KBBX	KBBX(N)	Kimberlite and country rock breccia	Mined out
	WBBX	WBBX(N)	Weathered country rock breccia	Mined out
	WK	WK(N)	Weathered kimberlite	Mined out
	FK(N)	FK(N)	Fragmental kimberlite	

Notes:

*These minor units were previously included in the major domain models and have now been modelled as separate domains.

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

The historical term 'breccia' meaning country rock xenolith-rich has been maintained for continuity with previous reporting.

Source: SRK (2025)

7.3.1 Units Defined by Weathering and Country Rock Dilution

The upper parts of each lobe consisted of calcretized and weathered horizons that are now mined out. The kimberlite units in these zones (CKIMB and WK) were classified based on their alteration and weathering characteristics which obscure the primary features. Each lobe also contained zones of high country rock dilution (basalt 'breccias') - in the South Lobe they were largely restricted to the upper weathered, now-depleted portion of the pipe, whereas in the Centre and North Lobes they extended to greater depths and have been entirely or largely mined out. These highly diluted zones comprised either brecciated country rock blocks with minor matrix kimberlite or zones of high basalt xenolith content within the pipe. The geological domain models representing these units were separated by lobe (Table 7-4).

Calcretized Kimberlite (CKIMB)

The upper 10 – 20 m of all three lobes comprised severely calcretized and silcretized rock. 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.

Basalt Breccia (BBX/KBBX)

Discontinuous zones of brecciated basalt (BBX) mixed with variable but generally minor amounts of kimberlite (typically less than 10%) were present in each of the lobes; they consisted of metre-sized to smaller basalt clasts set in a matrix of kimberlite and most occurred close to the wall-rock contact. An additional unit (KBBX) was defined for very basalt-xenolith kimberlite that appears broadly similar to BBX but displays lower levels of country rock dilution (50 to 90%). KBBX occurs interbedded and/or spatially associated with BBX, and Tait and Maccelari (2008) interpreted these as either talus-type slump deposits or as deposits of possible pyroclastic origin (given their higher kimberlite content relative to BBX). The BBX/KBBX units are now mined out in the South and Centre Lobes but extend below the current mining level in the North Lobe.

7.3.2 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 been mined out. Below the weathered and basalt breccia units, two volumetrically dominant kimberlite units, M/PK(S) and EM/PK(S), have been recognised, as well as eight volumetrically minor units, one of which (KIMB3) becomes more prevalent with increasing depth in the pipe.

The following descriptions of the M/PK(S), EM/PK(S), KIMB1 and KIMB3 units are restated from Nowicki et al. (2018) with additional information based on subsequent work (SRK, 2019, 2025) including (i) variations of the M/PK(S) and EM/PK(S) observed at depth in the pipe, (ii) updated description of KIMB3 based on improved understanding from 2018-2019 drilling, (iii) description of the other minor units, with updated description of WM/PK(S); and (iv) new features observed in M/PK(S) and EM/PK(S) in the pit.

Magmatic/Pyroclastic Kimberlite - M/PK(S)

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 7-9.

Summary: M/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly homogeneous to locally crudely stratified macrocrystic coherent or apparent coherent kimberlite.

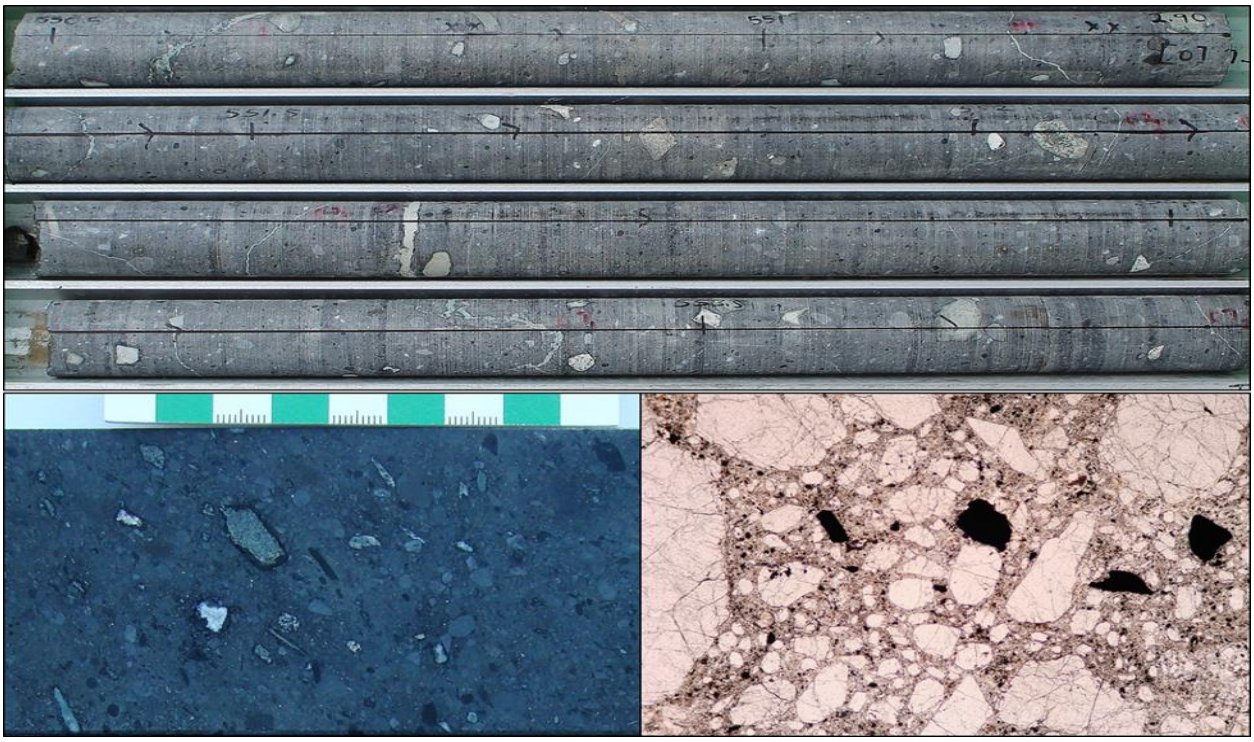
Key Characteristics:

- 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;
- Although M/PK(S) is broadly homogeneous, crude stratification in the form of diffuse variation 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 alteration halos and minor Karoo sedimentary rocks. Total country rock dilution is typically low (<10%), rarely ranging to a maximum of 25%, and most xenoliths are <10 cm in size;
- Ilmenite is notably abundant and characterised by variably developed grey reaction rims (comprising fibrous kelyphite-like material). In addition to garnet, ilmenite and rare chrome diopside, 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 and MARID xenoliths are also observed;
- M/PK(S) is characterised by a relatively high magnetic susceptibility (19 to 35 x 10^{-7} SI);
- Well-developed low-angle discontinuities are observed in M/PK(S) in the pit (Figure 7-6) and are interpreted to be the result of relaxation along pre-existing lava flow contacts;
- Rare possible flow contacts observed in drill core exhibit truncation of alignment textures; and
- Blocky/tabular fractured zones are also observed in M/PK(S) in the pit (Figure 7-10) and resemble similar features recorded in the entablature zone of basalt flows (Section 8).

Interpretation: The high abundance and inhomogeneous distribution of olivine and high proportion of angular broken olivine crystals, combined with the localised 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 coherent or locally apparent coherent (clastogenic) texture. Such kimberlites are believed to form by a range of processes which include effusive lava flows and lava fountain eruptions within an open diatreme or crater setting. The possible lava flow boundaries observed in the pit support this interpretation.

Figure 7-9: 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)

Figure 7-10: Blocky Fractures in M/PK(S)



Notes:
Scale – field of view approximately 1.5 m.
Source: SRK (2025)

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

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 7-11.

Summary: EM/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly homogeneous to locally crudely stratified macrocrystic coherent or apparent coherent kimberlite. It is distinguished from M/PK(S)

based on differences in olivine size distribution, primary groundmass mineralogy, and country rock xenolith and mantle xenocryst populations, amongst other features.

Key Characteristics:

- 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 variation 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, and interstitial serpentine/chlorite ± carbonate in differing relative abundances to those in M/PK(S). 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;

- 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 includes common clinopyroxene. Total country rock dilution is typically low (<15%), ranging to a maximum of 25%, and most xenoliths are < 10 cm in size;

- As in M/PK(S), ilmenite is characterised by variably developed reaction rims, but its abundance is roughly half that of M/PK(S). Orthopyroxene xenocrysts are more common than in M/PK(S) with less well-developed reaction rims;

- 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);

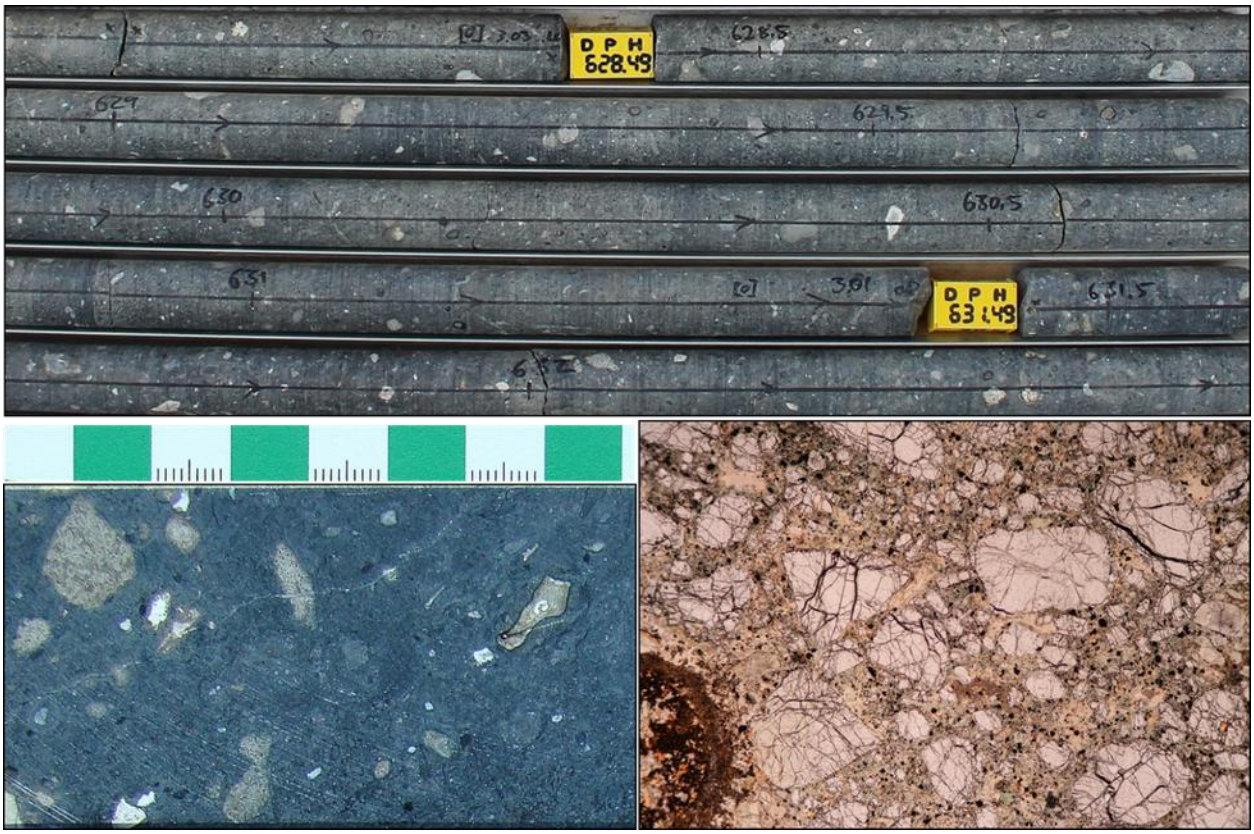
- Peridotite and eclogite xenoliths are present throughout;

- EM/PK(S) generally has a lower magnetic susceptibility than M/PK(S) (1.5 to 14 x 10^{-7} SI); and

- Irregular columnar jointing (Figure 7-12) is commonly observed in EM/PK(S) in the pit and pillow-like features are present locally.

Interpretation: The high abundance and inhomogeneous distribution of olivine and high proportion of angular broken olivine crystals, combined with the localised presence of crude stratification, rare probable relict melt-bearing pyroclasts, columnar jointing and pillow-like structures suggest EM/PK(S) was formed extrusively, and can be described as having a coherent or apparent coherent (clastogenic) texture. Such kimberlites are believed to form by a range of processes including effusive lava flows and lava fountain eruptions within an open diatreme or crater setting.

Figure 7-11: 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)

Figure 7-12: Irregular Columnar Jointing in EM/PK(S)



Source: (a) Alan Guest (2019); (b) SRK (2024)

Minor Unit KIMB3

KIMB3 was identified during core logging and petrographic studies undertaken on the South Lobe since 2017 (MSC, 2018b; SRK, 2019, 2025). Although a volumetrically minor component (<5%) of the total unweathered South Lobe infill, 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, KIMB3 intercepts are more dispersed, occurring along pipe contacts, internal contacts and randomly within the main units; previously these were incorporated into the surrounding M/PK(S) and EM/PK(S) domains in the geological model but have now also been modelled separately. Typical KIMB3 is shown in core, polished slab and photomicrograph in Figure 7-13.

Summary: KIMB3 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor, massive macrocrystic hypabyssal kimberlite.

Key Characteristics:

- In drill core, KIMB3 is dark grey-green in colour and characterised by readily discernible altered olivine (typically with dark margins) ranging in size to ultra coarse (> 16 mm). Olivine distribution is more uniform than in M/PK(S) and EM/PK(S) and broken crystals are rare. Olivine macrocryst abundance is lower than in 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 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 and typically completely altered;

- Autoliths of M/PK(S) and EM/PK(S) and others of unknown origin occur locally;

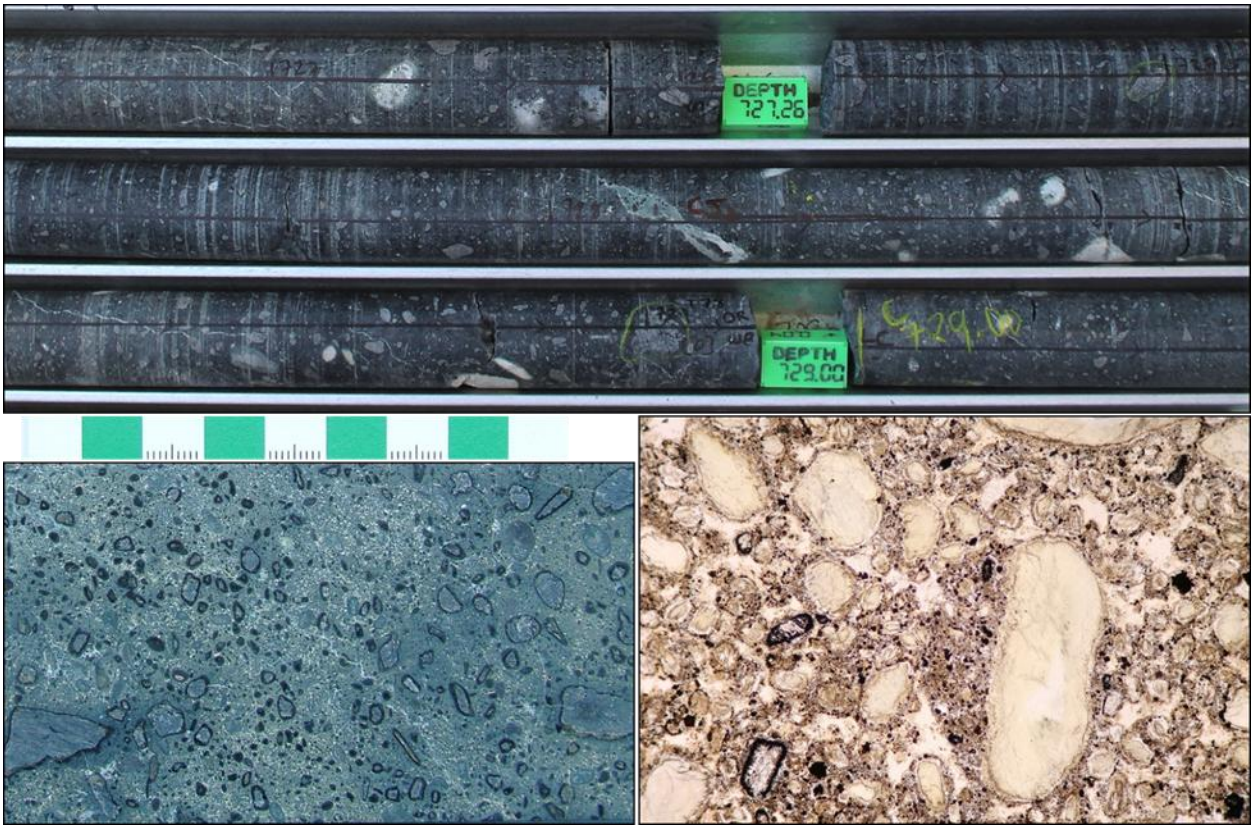
- Magnetic susceptibility readings for KIMB3 are highly variable but in general are the highest of all the units, commonly ranging between 20 and 60 x 10^{-7} SI;

- 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; and

- KIMB3 has not been observed in the pit to date.

Interpretation: 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.

Figure 7-13: 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 studies undertaken on the South Lobe since 2017 (MSC, 2018b; SRK, 2019, 2025). It is a volumetrically minor component (<1%) of the total unweathered South Lobe infill and generally occurs as discontinuous and dispersed intercepts along pipe contacts, internal contacts and apparently randomly within the main units, in some cases spatially associated with KIMB3. KIMB1 intercepts were previously incorporated into the M/PK(S) and EM/PK(S) domains in the geological model but have been modelled separately in the current updated model. KIMB1 has not been identified in the pit to date.

Summary: KIMB1 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor massive to locally flow-aligned macrocrystic hypabyssal kimberlite.

Key Characteristics:

- 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 phlogopite as ultra fine-grained tablets, 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);

- Magnetic susceptibility readings are highly variable but most commonly < 20 x 10^{-7} SI; and

- 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, suggesting KIMB1 represents low-volume late-stage sheet intrusions emplaced into the main pipe-filling units, possibly in some cases before they were completely consolidated.

Other Minor South Lobe Kimberlite Units

Six additional minor kimberlite units have been identified in the South Lobe: KIMB4a, KIMB5, KIMB6, WM/PK(S), RFW-ALT and KIMB7, which together make up a volumetrically minor part (<3%) of the pipe volume (Section 7.4.2).

KIMB4a: 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, a 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.

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. KIMB5 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.

WM/PK(S): Forms a pipe-shaped unit within M/PK(S) in the western part of the South Lobe. It is internally complex, both texturally and in terms of variability in country rock xenolith abundance, which ranges from <10 to 70%. Basalt is the dominant country rock xenolith and ranges widely in size from < 1 to > 100 cm. Most of WM/PK(S) is characteristically brown, highly altered, very basalt-rich and magmaclastic with a serpentine/carbonate matrix. The basalt and lesser basement xenoliths are rounded and large mantle xenoliths are common in parts. In the pit this material forms a vertical pipe-like zone that splits into three 'chimneys' in the upper benches and is interpreted as a gas escape structure. The remainder of WM/PK(S) comprises grey-blue, coherent to variably volcaniclastic kimberlite with a higher basalt content and less garnet than the surrounding M/PK(S). Historically WM/PK(S) was shown to differ in whole rock geochemistry, percentage DMS yield and rock density relative to EM/PK(S) and M/PK(S).

RFW-ALT: Comprises highly altered kimberlite of unknown type(s) and is typically associated with KIMB1 and KIMB3 along the pipe contacts, mainly in the northwest and south above 600 masl. The serpentine-clay alteration has almost entirely destroyed original textures and minerals. Due to the impact of alteration on the geotechnical properties of this material (drill core is broken or disaggregated) this unit has been modelled separately in the updated model.

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.

7.3.3 Centre and North Lobe Kimberlite Units

The Centre Lobe is infilled by kimberlite that bears a superficial resemblance to kimberlite in the North Lobe in that both lobes comprise non-fragmental, apparent coherent (historically magmatic) material as well as volcaniclastic (historically fragmental) kimberlite (Hanekom et al., 2006). Colour and texture variations are common in the Centre Lobe, and contacts between texturally distinct zones are generally gradational. In both lobes, the kimberlite textures locally alternate between apparent coherent and volcaniclastic. 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 recognised in the Centre Lobe and one in the North Lobe.

Carbonate-Rich Fragmental Kimberlite - CFK(C)

The now-depleted fresh infill in the upper part of the Centre Lobe comprised a fine- to coarse-grained olivine-rich, matrix-supported, poorly sorted and massive, carbonate-rich volcaniclastic to coherent kimberlite. Basalt is the dominant country rock xenolith with lesser basement and Karoo sedimentary rocks. Carbonate infilling of void space is common. Point counting data reported by Hanekom et al. (2006) on a very limited sample suite suggest that the carbonate-rich fragmental kimberlite unit (CFK(C)) generally contains higher concentrations of olivine

macrocrysts and lower country rock xenolith concentrations than the fragmental kimberlite unit (FK(C)). 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 to apparent coherent 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 characterised by the presence of large blocks of basalt. Basalt breccia (BBX) units in the Centre Lobe occur within the FK(C) unit rather than in the CFK(C) unit.

Fragmental Kimberlite - FK(N)

The remainder of the North Lobe consists of light greenish-grey, fine- to coarse-grained olivine-rich, matrix-supported, poorly sorted, massive volcaniclastic to apparent coherent 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 North Lobe kimberlite: rocks with a matrix consisting of both serpentine and calcite, and rocks with a matrix consisting predominantly of serpentine with minor calcite. No clear spatial distinction between the two groups could be resolved and the kimberlite was modelled as a single domain.

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 model was generated using Seequent's Leapfrog Geo software.

The pipe shell model has been updated in September 2025 for mining exposures of the contacts (all lobes) since the last update in 2019 (SRK, 2019; Doerksen et al., 2019, 2024) and at depth in the South Lobe using new pierce points from the 2025 core drilling program.

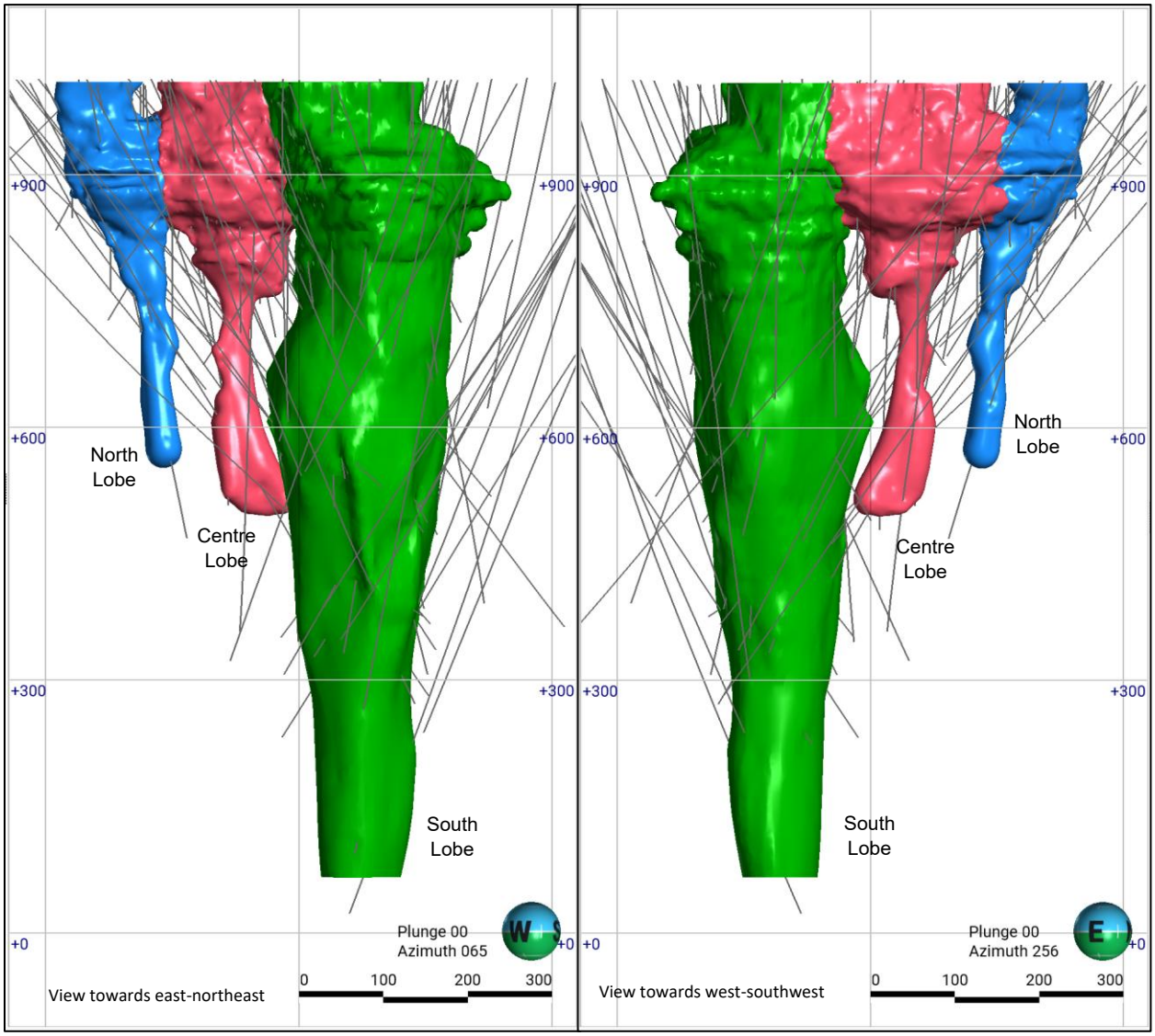
The South Lobe internal domain model has also been updated. In 2019, the main updates were: (1) a change in shape and decrease in size of the M/PK(S) domain below 500 masl and (2) generation of a new domain solid representing the distribution of the KIMB3 unit below 550 masl. The other minor kimberlite units described above were not modelled as discrete domains in 2019. In the current update, the minor units have been modelled separately. The internal domain model for the Centre and North Lobes remains unchanged from that documented in Oberholzer et al. (2017) and mining of these deposits has now ceased.

7.4.1 Shell Model

In 2019, the pipe shell model was updated for mining gains in all three lobes. Mapping of the pipe contact in the open pit since 2019 has enabled continued refinement of the model in the mining zone (above ~715 masl). Previous mapping in the South Lobe 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 (Figure 7-14). Similar pipe geometry was observed in the Centre and North Lobes, where the volume increases occur from 70 to 165 m below surface (930 to 845 masl) and are similarly constrained below by drilling. These zones have been mined out.

The undepleted pipe shell model (all lobes) is defined by a total of 174 pierce points in 101 core drillholes and an additional 15 pierce points in 13 LDD holes. The South Lobe model extends from surface (~1000 masl) to a minimum elevation of 66 masl (Figure 7-14) and is defined by 94 pierce points. The 2025 core drilling in the South Lobe provided an additional seven pierce points, resulting in minor or no shifts in the previously modelled pipe contact. The 2018-2019 core drilling provided 24 pierce points in 13 core drillholes in the South Lobe. The substantial internal and external (country rock only) drill coverage provides additional guidance on the minimum and maximum shell constraints respectively. 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 7-14: AK6 Pipe Shell Model



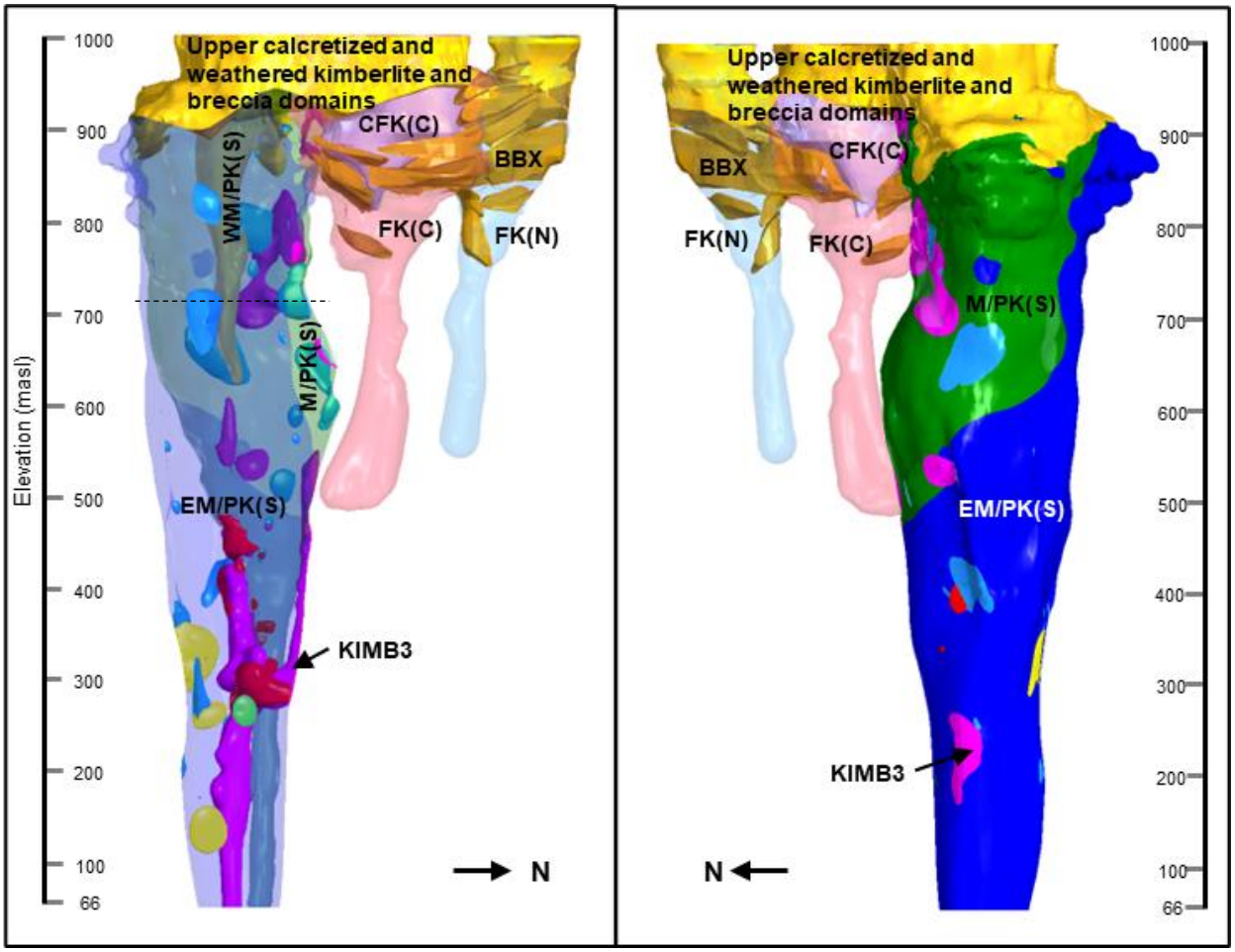
Notes: Shows all drillholes (black traces) used to define the model.
Source: SRK (2025)

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 7-4). The internal geological domains are shown in Figure 7-15 and the number and length of core drillholes defining each domain are given in Table 7-5. In total there

has been approximately 15,000 m of core drilling in the unweathered South Lobe. M/PK(S) and EM/PK(S) are the two major domains in the South lobe. The geometry of the domains and the steep, curved contact between them indicates that EM/PK(S) predates M/PK(S).

Figure 7-15: Internal Geological Domains of the AK6 Kimberlite



Notes:
The upper ~70 to 100 m of calcretised and weathered kimberlite and country rock breccia units which are completely mined out are shown in a single colour to simplify the figure. Some domains are rendered transparent to display the internal domains. Minor units: KIMB1 (light blue); KIMB4a (red), KIMB5 (yellow), KIMB6 (pale green), WM/PK(S) (orange). The dashed line at ~715 masl shows the elevation of the pit surface in the South Lobe at the end of September 2025.
Source: SRK (2025)

Table 7-5: Core Drill Coverage of Internal Geological Model Domains

Lobe	Domain	Number of Core Holes	Drillhole Intersection Length (m)
South	BBX(S), CKBBX(S), CKIMB(S), WBBX(S), WKBBX(S), WK(S), IntSWBas	31	2,023.40
	M/PK(S)	55	8,242.00
	EM/PK(S)	48	5005.45
	KIMB3	18	626.90
	KIMB1	28	354.40
	KIMB4a	6	162.12
	KIMB5	5	85.90
	KIMB6	2	11.00
	WM/PK(S)	7	405.20
	RFW-ALT	18	104.8
Centre	BBX(C), CKIMB(C), KBBX (C), WK(C)	20	1,264.90
	CFK(C)	18	1,047.70
	FK(C)	25	1,272.00
North	BBX(N), CKIMB(N), WBBX(N), WKBBX(N), WK(N)	13	914.60
	FK(N)	14	1,008.40

Source: SRK (2025)

The position and geometry of the M/PK(S)-EM/PK(S) contact has been refined based on pit mapping since 2019 and the 2025 drilling. The M/PK(S) and EM/PK(S) model solids were last updated in 2019, the most significant change being below 500 masl, where the 2018-2019 drilling indicated a decrease in the previously 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.

The M/PK(S) domain remains poorly constrained below 440 masl in the current updated model as it is not directly drill-supported 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 7-16). 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 suggested the KIMB3 unit is most common. The 2025 drilling has resulted in minor refinement of the KIMB3 model. As described in Section 7.3, KIMB3 is a hypabyssal kimberlite

that post-dates and intruded into the M/PK(S) and EM/PK(S). It becomes more prevalent below 550 masl, and 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.

Previously, the minor kimberlite units other than KIMB3 were not resolved as discrete domains due to their small volumes and generally discontinuous and dispersed distributions along pipe contacts, internal contacts and within the M/PK(S) and EM/PK(S) and hence were left as formally undefined minor volumes within the main domains. In the current updated model, the other minor units have – like KIMB3 – been modelled separately to enable their volumes to be estimated and to improve knowledge of their geotechnical parameters.

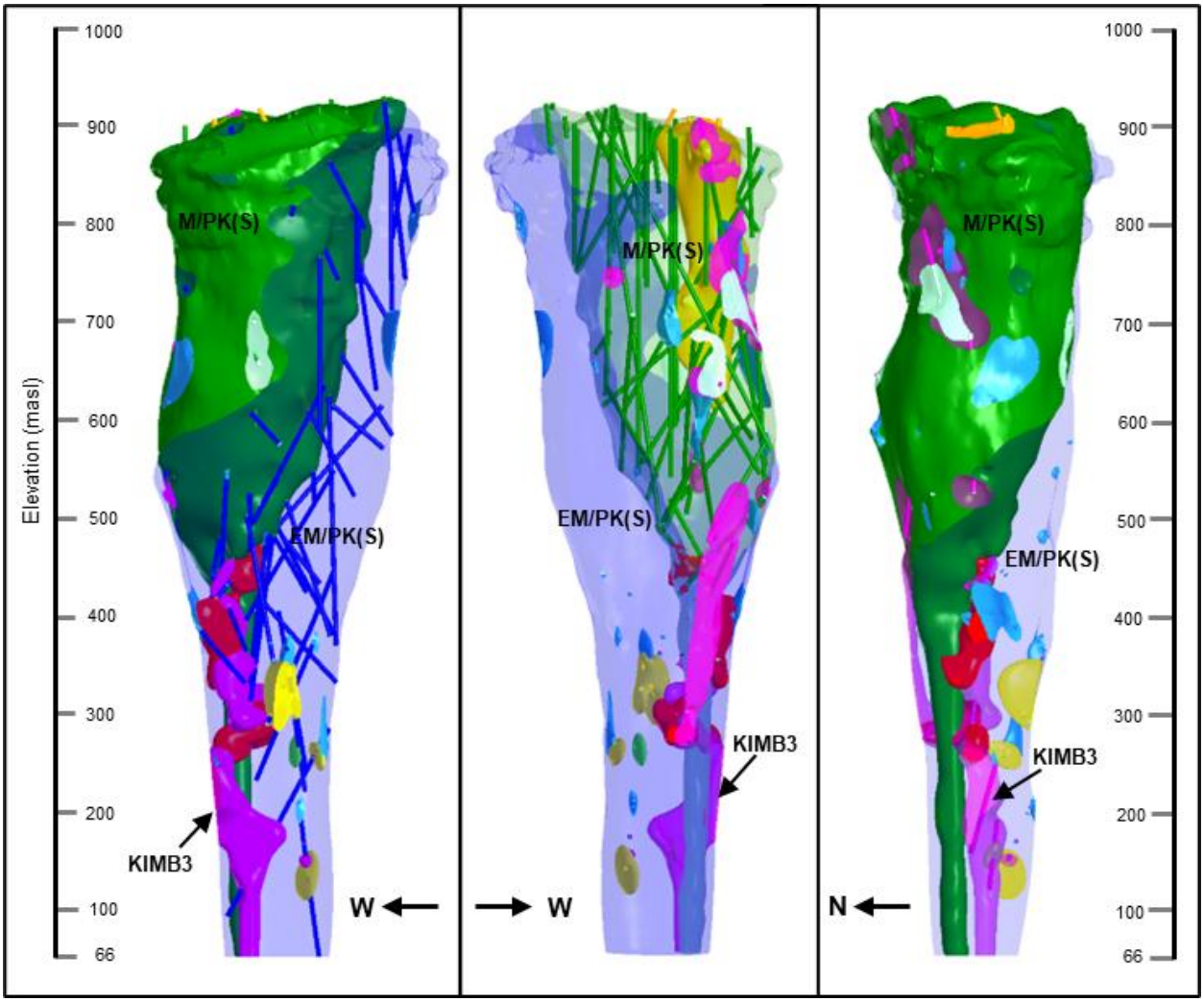
The volumes of the South Lobe kimberlite domains below September 30, 2025 pit surface in depth intervals corresponding to the Indicated and Inferred Resources are shown in Table 7-6. The morphology of the domains and their internal drill coverage are illustrated in Figure 7-16.

Table 7-6: Volume Estimates of South Lobe Internal Domains (below September 30, 2025 pit surface)

Domain	Pit Surface* to 250 masl		250 to 66 masl		Total	
	Mm³	%	Mm³	%	Mm³	%
M/PK(S)	5.28	44	0.10	6	5.39	39.7
EM/PK(S)	5.97	50	1.40	85	7.36	54.2
KIMB3	0.31	3	0.14	8	0.45	3.3
KIMB1	0.07	1	0.00	0	0.07	0.5
KIMB4a	0.10	1	0.00	0	0.10	0.8
KIMB5	0.05	0	0.01	1	0.06	0.4
KIMB6	0.00	0	0.00	0	0.00	0.0
WM/PK(S)	0.13	1	0.00	0	0.13	0.9
RFW-ALT	0.02	0	0.00	0	0.02	0.1
All Domains	11.92		1.65			
Total	**13.57**					**100**

Note:
Base of OP at September 30, 2025 was at ~715 masl.
Base of Indicated Resource is at 250 masl; Base of model is at 66 masl.
Due to rounding some columns or rows may not compute exactly as shown.
Source: SRK (2025)

Figure 7-16: 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) as of September 2025 and the internal core drill coverage used to define them. Minor units: KIMB1 (light blue); KIMB4a (red), KIMB5 (yellow), KIMB6 (pale green), WM/PK(S) (orange).

Source: SRK (2025)

7.4.3 Geological Continuity

Demonstration of geological continuity within the main kimberlite units is required for the Mineral Resource Estimate to permit assignment of average diamond grades and values (prices) derived from production data to kimberlite at depth (Section 14). A thorough assessment of the degree of geological continuity was carried out by MSC in support of the 2018 resource update (Nowicki et al., 2018). This involved review of surface exposures, drill cores and dilution measurements, and an extensive petrographic study. The 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. The QP carried out much of this work while employed at MSC and has subsequently continued to assess the degree of continuity within the kimberlite units during work conducted since 2018.

Surface and Drill Core Observations

Historical AK6 geology reports do not indicate any major geological discontinuity with depth within the volumetrically dominant kimberlite units. Kimberlite exposures in the OP have been regularly examined by the QP. 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 geological model update, a detailed review of 13 of the 2018-2019 drill cores was undertaken on site in May 2019, and the 2025 drill cores were reviewed in photographs and on site. The observations did not highlight any major features or changes in the size and abundance of macroscopic constituents in 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.

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). 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 localised 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) (MSC, 2018b; 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 Estimates)

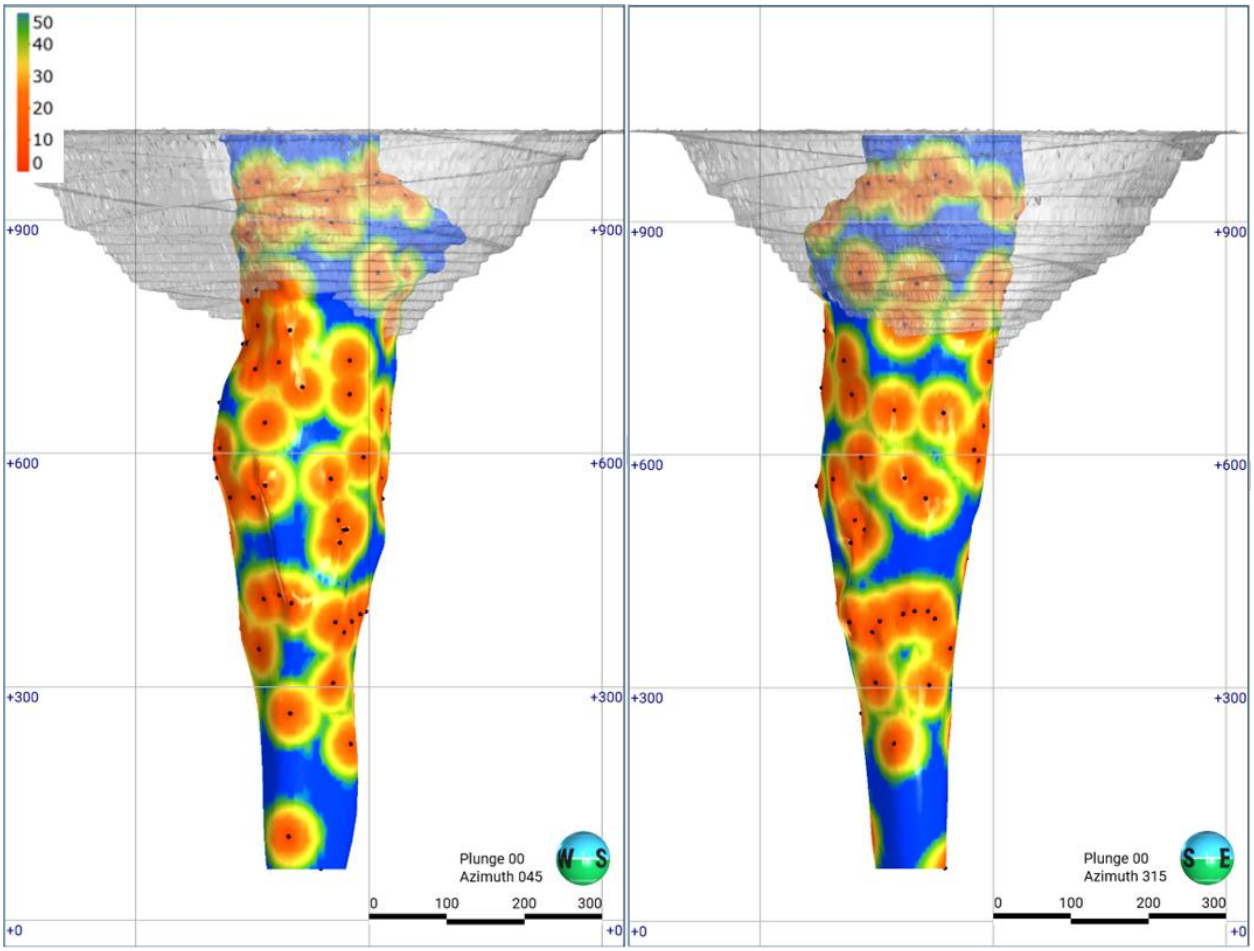
The AK6 pipe shell model is constrained by 189 pierce points from 101 core and 13 LDD drillholes, the majority of which intersect above 500 masl. The model is well constrained in this upper zone by these pierce points and extensive internal coverage providing minimum constraints on the size of the body.

The September 2025 South Lobe pipe shell model is well constrained by 52 pierce points above 600 masl and by 25 pierce points between 600 and 400 masl. The 2025 drilling provided an additional seven pierce points in the South Lobe, six above and one below 400 masl. The model is less well constrained by 13 pierce points between 400 and 250 masl. However, while there is scope to modify the exact position of the contact in the gaps between pierce points in this elevation range (Figure 7-17), 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 from 870 to 250 masl. 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 22,788 m of internal core drilling, of which 17,280 m occurs in the South Lobe. The degree of control on the boundaries between the main South Lobe internal domains M/PK(S) and EM/PK(S) is relatively high between surface and ~450 masl. However, there is only a single intersection of M/PK(S) below 440 masl and its volume below this elevation is thus largely constrained by reasonable internal drill coverage, including intercepts of EM/PK(S), KIMB3 and other minor units, which confirm where MP/K(S) is not present. The currently modelled distribution of KIMB3 likely represents a minimum volume for this unit. Minor units KIMB1, KIMB4a, KIMB5 and KIMB6 comprise multiple narrow, spatially separate occurrences, each constrained by only one or two drillholes and their modelled discontinuous distributions also likely represent minimum volumes. The lower confidence in the volumes of the minor units and lack of macrodiamond sampling are factors considered in resource classification (Section 14).

In the 2018 model, the uncertainty in Mineral Resource Estimates below 400 masl was 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. In the 2019 model, the additional drill coverage and microdiamond sampling from the 2018-2019 drilling provided a basis for upgraded confidence between 400 and 250 masl (excluding KIMB3 domain). In the 2025 model, the pierce points and internal coverage from the five 2025 drillholes have served to further refine the pipe shell model and internal domain boundaries with an overall 2% decrease in volume from the 2019 model in the zone below the end-September 2025 pit surface and the base of the model.

Figure 7-17: Drillhole Pierce Points in the South Lobe



Note:
Drillhole pierce points (black dots) in the South Lobe (left, view looking northeast; right, view looking northwest) with distance contours. Blue areas are > 50 m from pierce points. September 30, 2025 pit surface.
Source: SRK (2025)

7.4.5 Emplacement Model

Many kimberlite pipes, including those at the Orapa and Letlhakane Mines 20 km to the north and northeast, are infilled by a variety of textural kimberlite types including pyroclastic, hypabyssal and resedimented volcaniclastic kimberlite. The AK6 kimberlite, in particular the South Lobe, differs in being comprised entirely of coherent or apparent coherent kimberlite. The Centre and North Lobes display more textural variability from coherent to lesser volcaniclastic kimberlite. AK6 is similar in age to Orapa and Letlhakane (and other kimberlites in the OKF) and occurs at similar elevations to the volcaniclastic-filled pipes. None of the kimberlite at AK6 resembles the type of pyroclastic kimberlite present in the Orapa and Letlhakane pipes (termed KPK; Scott Smith et al., 2013; 2018).

The regular morphology of the South Lobe, the presence of coherent kimberlite over a significant vertical extent (900 m drill confirmed) and the characteristics of the two main pipe-filling coherent kimberlite units, M/PK(S) and EM/PK(S), indicates that they formed extrusively by emplacement of effusive lava flows and/or lava fountains into the previously excavated pipe to form lava lake deposits. The M/PK(S) and EM/PK(S) were later intruded by various small-volume phases of hypabyssal kimberlite, mainly along internal and pipe contacts.

The characteristics of the M/PK(S) and EM/PK(S) that support this interpretation and are relevant to 3D modelling, resource estimation and mining of the South Lobe are as follows:

- Presence of two distinct phases (M/PK(S) and EM/PK(S)), each of which is continuous over significant vertical depths with sharp or gradational sub-vertical contacts between them, and that are broadly homogeneous but exhibit variations in the size distribution and relative proportions of components over several metres to tens of metres vertically and laterally, suggestive of crude layering or stacked individual flows;

- Presence of sub-horizontal discontinuities or contacts between thick vertical packages of broadly homogeneous coherent kimberlite, in some cases displaying truncation of alignment textures at the contacts, which are interpreted as flow boundaries;

- Very high total olivine abundances, high proportions of broken olivine and inhomogeneous olivine distribution, which notably contrasts with the olivine in the late-stage HK intrusions;

- Low country rock xenolith (dilution) contents (typically <15%), which are higher than in the late-stage hypabyssal intrusions (minor units) and notably lower than dilution in the pyroclastic kimberlites at Orapa and Letlhakane (typically 25-50%);

- Local rare occurrences of magmaclasts with diffuse margins that are interpreted as relict melt-bearing pyroclasts;

- Variably developed preferred alignment of components, mainly in M/PK(S);

- Presence of irregular columnar jointing (Figure 7-12), pillow structures, and tabular/blocky fractured zones (Figure 7-10) displaying similarities to features in the entablature zones of some basaltic lava flows (Forbes et al., 2014; Philips et al., 2013); and

- The cross-cutting hypabyssal kimberlite sheets are texturally distinct from and locally contain clasts (autoliths) of the M/PKS or EM/PKS through which they intruded. Contacts are variably sharp to diffuse with finer-grained flow zones developed in some cases.

The geometry and distribution of M/PK(S) and EM/PK(S) indicate that EM/PK(S) was emplaced first into an excavated pipe/crater, followed by emplacement of M/PKS after the partial excavation of EM/PK(S) by one or more explosive eruptive events. Whereas the pipe infilling processes are interpreted to be low energy or non-explosive (lava fountain or lava flow), the pipe excavation process must have been explosive. The high-energy explosive mechanisms are unknown, however water-magma interaction is likely if groundwater was circulating in country rock aquifers.

7.4.6 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.

7.5 Mineralization

KDM is one of the world's most significant producers of large and high-value diamonds including fine-gem Type IIa and rare coloured diamonds. From inception to the end of Q3 2025, Karowe has recovered 43 diamonds > 300 ct, 69 diamonds between 200 and 300 ct, and 283 diamonds between 100 and 200 ct. The mine has recovered nine diamonds in excess of 1,000 ct since 2015. The South Lobe has consistently recovered high-value diamonds in excess of 200 ct and several of the Legacy stones (Section 14) have come from the EM/PK(S), including the 1,109 ct Lesedi La Rona.

The M/PK(S) and EM/PK(S) each have distinct diamond grades, size frequency distributions (SFD), average US$/ct diamond values and significant stone recoveries (Section 14). The SFDs are considered as coarse with roughly 60 to 70% of the mine's revenue being generated by +10.8 ct diamonds (Specials) that make up greater than 6% of the carats produced.

Since 2012, the mine has produced over 290,000 ct and over 9,000 stones in excess of 10.8 ct for an average stone size of 30.9 ct/stn for the +10.8 ct production. In the period from 2015 to September 30, 2025 inclusive, the mine has averaged 6.4 wt% of total production being >10.8 ct in weight, and 7.5 wt% of M/PK(S) and EM/PK(S) production. The nine stones exceeding 1,000 ct include a 1,174 ct diamond recovered in 2021, a 1,080 ct diamond recovered in 2023, and the 2,488 ct Motswedi and 1,094 ct Seriti recovered in Q3 2024. Select legacy stones and an example of KDM run-of-mine diamonds are shown in Figure 7-18.

Diamonds at Karowe are classified into four primary mantle source suites based on the chemical composition of their mineral inclusions (Motsamai et al., 2018): 53% of Karowe inclusion-bearing diamonds are derived from eclogitic sources, 44% are peridotitic, 2% have a sublithospheric origin, and 1% are websteritic. A sublithospheric inclusion suite was established by eclogitic garnets containing a majorite component, a feature so far unique within the Orapa kimberlite cluster (Motsamai et al., 2018). The origin of the majorite-bearing diamonds is estimated to be between 330 to 420 km depth, straddling the asthenosphere–transition zone boundary. The observation of superdeep mineral inclusions in Karowe diamonds is consistent with a sublithospheric origin for the exceptionally large diamonds.

Figure 7-18: Karowe Run-of-Mine and Select Legacy Diamonds



Notes:

(a) 1,109 ct Lesedi La Rona, recovered in 2015, on a sample of its kimberlite host, EM/PK(S); (b) 813 ct Constellation, also recovered in 2015 from EM/PK(S); (c) 62 ct pink Boitumelo; (d) 549 ct Sethunya recovered in 2020; (e) 2,488 ct Motswedi recovered in 2024; (f) run-of-mine parcel (g) 242 ct and 216 ct recovered in 2022.

Source: Lucara (2025)

8 DEPOSIT TYPES

The primary source rocks for diamonds currently being mined or that have been mined worldwide are kimberlites, orangeites and olivine lamproites. The vast majority of primary diamond deposits mined globally consist of steep-sided, pipe-shaped bodies infilled with volcaniclastic kimberlite (Scott Smith et al., 2013, 2018). Less common are diamond mines hosted in hypabyssal kimberlite and orangeite dykes or sills (intrusive sheets) and in pipes comprised entirely of coherent kimberlite.

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

Kimberlite deposits are typically formed from multiple batches of magma that were emplaced at or near the earth's surface. The total or remnant emplacement products of a single batch of magma is termed a phase of kimberlite (Scott Smith et al., 2018). Each phase of kimberlite typically has distinct diamond populations, grades, and values. Most kimberlite bodies comprise more than one phase of kimberlite, and the internal architecture of the various phases and nature of contacts between them can vary considerably within and between deposits. Development of a robust geological model and lithologically controlled sampling are thus important in evaluation.

The emplacement of kimberlite at or just below the surface of the crust as sheet-like or irregular intrusions or as volcanic pipes (Figure 8-1) is influenced by many factors (Clement and Reid, 1989; Field and Scott Smith, 1999; Sparks et al., 2006; Barnett, 2008) which include the following:

- Magma characteristics (volatile content, viscosity, crystal content, volume, temperature);
- Nature of the host country rocks;
- Local structural setting;
- Local and regional stress fields; and
- Presence of water.

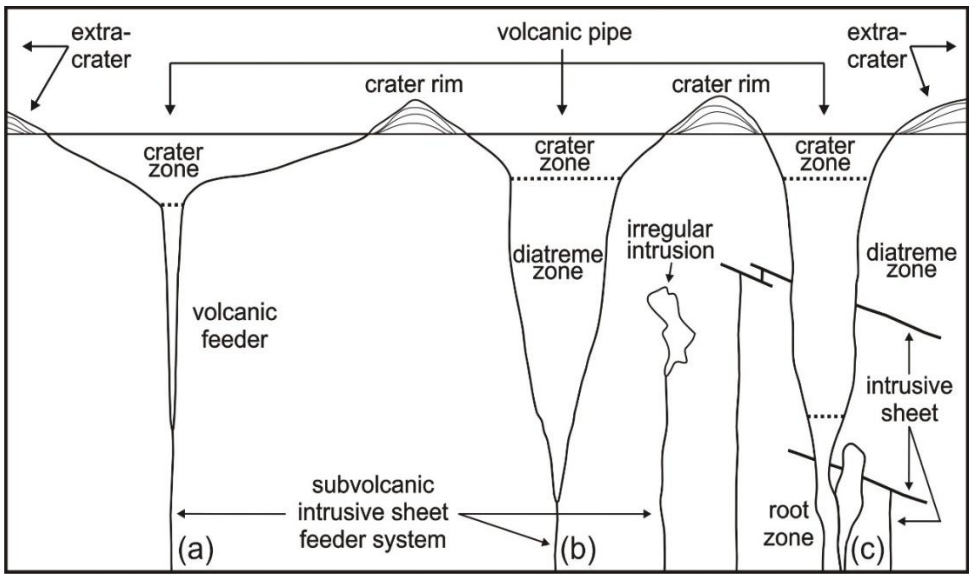
Sheets and irregular intrusions are typically emplaced along pre-existing planes of weakness in the country rock. Their emplacement does not involve explosive volcanic activity, and they typically consist of texturally unmodified hypabyssal kimberlite, a variety of coherent kimberlite. Volcanic pipes are generated by explosive volcanic activity (related to degassing of magma and/or interaction of magma with water) which produces clasts of the kimberlite magma (melt-bearing pyroclasts; Scott Smith et al., 2013) and of the country rock into which it was emplaced (country rock xenoliths). Deposits formed by explosive 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 and varying properties of the kimberlite magma, kimberlite volcanoes can take a wide range of forms and be infilled by a variety of deposit types (Skinner and Marsh, 2004; Scott Smith, 2008). Kimberlite pipes are often highly eroded with only the deeper portions preserved; crater deposits occur at some localities and extra-crater deposits are rare. Kimberlite volcanoes are infilled by a 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). Not all kimberlite volcanoes are filled with volcaniclastic kimberlite. Some pipes are filled with coherent kimberlite deposits formed extrusively by emplacement of lava flows and/or lava fountains (see below) into previously excavated pipes or craters to form lava lake deposits.

Coherent (previously termed magmatic) kimberlites are the products of direct crystallization of kimberlite magmas and typically comprise of olivine set in a fine-grained crystalline groundmass made up of serpentine and/or carbonate as well as varying amounts of phlogopite, monticellite, 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 (macrocrysts).

In addition to mantle-derived olivine, kimberlites also commonly contain other mantle-derived minerals, the most common being garnet, chrome diopside, chromite and ilmenite. These minerals, referred to as kimberlite indicator minerals, are important for kimberlite exploration and evaluation as they can be used to find kimberlites (by tracing the indicator minerals in surface samples), to provide early indications of a kimberlite's potential to contain diamonds, and for discriminating between different kimberlite phases in a particular body.

Figure 8-1: Kimberlite Body Types and Morphology



Source: Scott Smith et al., (2013, 2018)

Diamonds are xenocrysts (meaning “foreign” crystals) within kimberlite as they are primarily formed and preserved in the deep lithospheric mantle (the oldest, thickest parts of continents) at depths of 150 – 200 km and 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. The diamonds at Karowe include a population of ‘super-deep’ (sublithospheric) diamonds classified as Type II diamonds (see below) that originate from depths greater than 300 km.

Diamonds occur in kimberlite in trace amounts (ppm) as a dispersed particulate mineral. They can vary significantly within and between different kimberlite deposits in terms of their total concentration (grade in carats per hundred tonnes), particle size distribution and physical characteristics (e.g., colour, shape, clarity and surface features).

The value of each diamond and thus the average value of each diamond population is governed by the size and physical characteristics of the stones. The concentration of diamonds in any given kimberlite is dependent on the following factors:

- The extent to which the source magma has interacted with and sampled potentially diamondiferous deep lithospheric mantle;
- The diamond content of the sampled mantle;
- The extent of resorption of diamond by the kimberlite magma;
- Physical sorting and/or winnowing processes during volcanic eruption and deposition; and
- Dilution of the kimberlite with barren country rock or surface sediments.

At Karowe, the extent of mantle sampling and type of mantle sampled are considered to be the main factors controlling variation in diamond grade between phases.

The diamond size distribution characteristics of a kimberlite deposit are inherited from the original population of diamonds sampled from the mantle, but they can be affected by secondary processes such as resorption during magma ascent and sorting during eruption and deposition.

The physical characteristics of diamonds in a kimberlite deposit are largely inherited from the original characteristics in their mantle source rocks but can be affected by processes associated with kimberlite emplacement, such as:

- 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 immediately prior to or at the early stages of kimberlite emplacement; and
- Physical breakage of the diamonds during turbulent or explosive emplacement processes.

The primary difference between Type I and Type II diamonds is the presence or absence of measurable nitrogen impurities in their crystal structure:

- **Type I**: Diamonds contain nitrogen and are the most common (~98% of natural diamonds). The nitrogen affects their colour, often giving them a yellow or brown tint; and

- **Type II**: Diamonds have no detectable nitrogen and are considered the most chemically pure expression of natural diamonds. They are much rarer (1-2% of natural diamonds) and often colourless, though they can also occur in rare fancy colors like pink or blue; the latter are characterised by the presence of boron and are extremely rare (Type IIb diamonds). Large, D-colour, flawless diamonds are almost always Type IIa diamonds.

Karowe Diamond Mine is a well-known locality where high-value, fine-gem Type IIa diamonds occur; they characteristically occur in the M/PK(S) and EM/PK(S) in the South Lobe though are not limited to these units, occurring in all three lobes of the deposit.

8.1 Conceptual Geological Model

Coherent kimberlite (CK) can occur as dykes and sills (sheet intrusions), irregular-shaped bodies separate from or associated with the root zones of diatremes, and as pipe-shaped bodies partially or completely infilling diatremes. CK that crystalizes as subsurface intrusions is referred to as hypabyssal kimberlite or intrusive CK (Scott Smith et al., 2013). The characteristics of pipe-filling CK or pfCK (Webb et al., 2008; Tovey et al., 2022) are attributed to extrusive emplacement as lava flows, lakes and/or fountains with the degree of olivine fragmentation used in conjunction with other characteristics to infer higher or lower energy emplacement processes (Tovey et al., 2022 and references therein). For all pfCK, the stage of pipe infilling by CK was preceded by more explosive activity resulting in pipe excavation, as evidenced by the low abundances of country rock xenoliths in pfCK within pipes without any volcaniclastic kimberlite (Scott Smith et al., 2018; Tovey et al; 2022).

Globally, extrusive coherent kimberlite deposits (kimberlite lavas) are not common and there are only a few known occurrences of diatremes or craters comprised entirely or predominantly of extrusive coherent kimberlite (e.g., Eley et al., 2008; Pell et al., 2013; van Straaten et al., 2011, Webb et al., 2008). Recognition of key diagnostic features of intrusive vs extrusive kimberlites has led to an increase in documented examples of extrusive CK. The pipe-filling coherent kimberlite at AK6 is broadly similar in terms of morphology, texture and fabric/structure to those described from the Canadian and Angolan localities. The M/PK(S) and EM/PK(S) units would be classified as very large volume pfCK deposits according to Tovey et al. (2022).

Basaltic systems provide a useful analogue for visualizing the processes interpreted to have been involved in the formation of pfCK and understanding the features observed in the South Lobe at Karowe (Figure 8-2, Figure 8-3, and Section 7.3).

A lava fountain is a continuous spray of disrupted magma erupting through a vent to form a persistent fountain of molten magma above the vent. The fountain is supported either by the hydrostatic pressure of magma in the upper part of the volcanic structure, or by expanding gas released from the magma during the eruption. Fall-out from the fountain produces a spatter rampart around the vent and if the accumulation rate is high the molten spatter may coagulate to form a flow of lava (a ‘clastogenic flow’; Allaby, 2008). Relict pyroclastic textures may be evident

such as diffuse melt-bearing pyroclasts, high proportions of broken crystals, and concentrations of crystal and/or lithic components.

Figure 8-2: Basaltic Lava Fountain and Lava Flows



Source: USGS, M. Patrick (2025)

Entablature is the term used to describe zones or tiers of irregular jointing in basaltic lava flows, including irregularly-orientated columnar joints and hackly fracturing/cube-jointing (Forbes et al., 2014). Entablature is found in basaltic lava flows in many places and is thought to form when water from rivers dammed by the lava inundates the lava flow surface and during meltwater-lava interaction in subglacial settings. The entablature features shown in Figure 8-3 bear resemblance to the irregularly-orientated columnar joints and hackly fracturing observed in the M/PK(S) and EM/PK(S) at Karowe.

Figure 8-3: Entablature Zone of Basaltic Lava Flow



Source: Forbes et al. (2014), left; Getty image, right

9 EXPLORATION

9.1 Historical Exploration: 2003 to 2008 – De Beers and Boteti JV

9.1.1 Background

This section summarises advanced exploration work (used to support resource estimates) on the AK6 kimberlite carried out by De Beers Prospecting Botswana (Pty) Ltd (Debot), the operator of the Boteti Joint Venture, under PL 13/2000. Previous work programs by De Beers prior to the joint venture are briefly summarised here (extracted 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.

The AK6 kimberlite was continuously held by De Beers under a succession of prospecting licences from the time of its discovery in 1969, until the Project was acquired by Lucara in 2009 (Section 6). 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 or that the resource was previously under-sampled.

9.1.2 Approach and Methodology

Exploration of the AK6 kimberlite by previous owners is shown in Table 9-1. It followed a staged approach, which can be summarised 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 ct from 13 large diameter drillholes (LDD) 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; and

- Advanced Exploration Phase 2 – The results of Phase 1 merited conducting Phase 2, the objective of which was to define an Indicated Mineral Resource and recover a large diamond parcel, ideally 3,000 ct, to reduce revenue uncertainty. LDD were placed at 50 m centres and trenches were excavated 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 Centre Lobes. These were considered to be the limits of possible OP mining based on an initial economic assessment;

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

Owner	Stage	Work Done	Duration
De Beers and Boteti JV	Early Evaluation	5 x 12¼" large diameter drillholes totalling 679 m, 97 t bulk sample	2003 - 2005
		DMS and diamond recovery	
		Geophysical surveys	
Boteti JV	Phase 1 Advanced Exploration	44 x 6½" percussion holes for delineation totalling 4,575 m	2005 - 2006
		12 x vertical core holes (NQ) as LDD pilots, totalling 2,980 m	
		17 x inclined core holes (NQ) for delineation totalling 6,901 m	
		13 x 23" LDD totalling 3,699 m	
		DMS processing and diamond recovery from 1,775 t	
Boteti JV	Phase 2 Advanced Exploration	11 x vertical core holes (NQ) as LDD pilots totalling 4,184 m	2006 - 2008
		29 x inclined core holes (NQ) for delineation totalling 8,679 m	
		12 x 23" LDD totalling 4,265 m	
		Trench bulk sampling at surface (refer to Figure 10-1)	
		DMS processing and diamond recovery from 2,235 t	

Source: Modified from McGeorge et al. (2010)

9.1.3 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.

Method	Line km	Comments
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: Modified from McGeorge et al. (2010)

9.2 Exploration by Lucara

No exploration other than core drilling has been carried out on the AK6 kimberlite by Lucara since acquiring the project in 2009. Lucara's core drilling and sampling programs are summarised in Table 9-3 and described in Section 10. The current resource estimate is based on data collected during these programs, incorporating results from mining operations and diamond sales since the commencement of commercial production in 2012.

Table 9-3: Summary of Lucara Exploration at AK6 – Core Drilling and Sampling

Program	Work Done	Duration
Delineation and Geotechnical Drilling	15 x core holes (HQ and NQ) totalling 12,272 m 916 microdiamond samples (7,315 kg)	2016 - 2017
Delineation and Geotechnical Drilling	37 x core holes (HQ and NQ) totalling 23,958 m 153 microdiamond samples (1,232.8 kg)	2018 – 2019
Delineation And Geotechnical Drilling	5 x core holes (HQ and PQ) totalling 1,781 m	2025

Source: SRK (2025)

10 DRILLING

10.1 Historical Delineation and Bulk Sample Drilling

Early drilling (2003 to 2008) of the AK6 kimberlite by De Beers and the Boteti JV is described in detail in a previous Technical Report (McGeorge et al., 2010). A brief summary is provided here, extracted from Oberholzer et al. (2017). The historical drilling can be assigned to three categories:

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

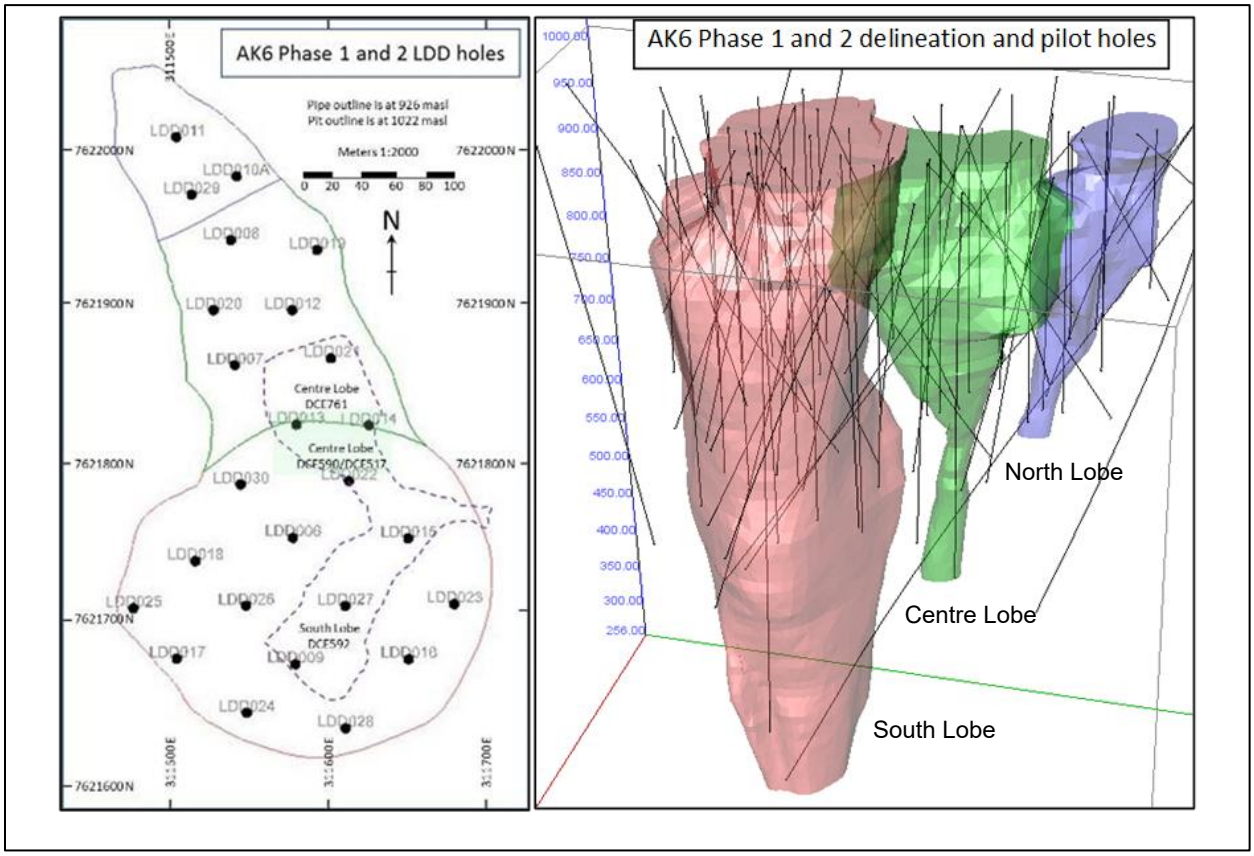
The historical drilling is summarised in Table 10-1, grouped into the exploration phases described in Section 0. The LDD, delineation and pilot hole locations are illustrated in Figure 10-1. Early evaluation holes are not shown as they were not used to support Mineral Resource Estimates.

Table 10-1: Historical Drilling at AK6 by De Beers / Boteti JV (2003 to 2008)

Exploration 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	6,901	2005
	Piloting	Core	NQ	12	2,980	2005
	Bulk sampling	LDD	23"	13	3,699	2005 - 2006
Phase 2 advanced exploration	Piloting	Core	NQ	11	4,184	2005 - 2006
	Delineation	Core	NQ	29	8,679	2006 - 2007
	Bulk sampling	LDD	23"	12	4,265	2006 - 2008

Source: Modified after McGeorge et al. (2010)

Figure 10-1: Historical Phase 1 and 2 Drilling



Notes:
LDD holes (left, plan view) are all vertical. The outline of a surface trench bulk sample is shown as a dotted black line. Core holes (right, inclined view of 2018 model towards the southwest) are shown as thin black traces.
Source: Nowicki et al. (2018)

10.2 Lucara Delineation and Geotechnical Drilling

Lucara has completed four core drilling campaigns as detailed in the sections below:

- In 2016 and 2017, two core drilling programs were conducted with a combined total of 15 holes and 12,272 m drilled to provide additional pierce points for pipe delineation and new geological and geotechnical information for the deeper portion (below 600 masl) of the South Lobe and the surrounding country rocks;
- In 2018 and 2019, a combined geotechnical and delineation drilling program was conducted with 37 holes and 24,000 m drilled. Some drilling was specific to the country rock and several holes were designed to test the South Lobe for geotechnical purposes, including at depths below 400 masl;

- In 2020, two vertical core holes were drilled in the country rocks adjacent to the South Lobe to collect geotechnical information ahead of shaft sinking, with a total of 1,515 m drilled; and

- In 2025, five core holes were drilled in the South Lobe to provide additional geological and geotechnical information for the minor kimberlite units and placement of UG infrastructure, with a total of 1,781 m drilled.

10.2.1 2016-2017 Drilling: South Lobe 400-600 masl Definition

Two core drilling programs were completed in 2016-2017 to support further evaluation of the deeper portion of the South Lobe between 400 and 600 masl (400 to 600 m below surface) and to provide geotechnical information on the host rock stratigraphy and physical properties. A total of 12,272 m was completed from 15 drillholes, as summarised in Table 10-2. Drill coverage is shown in Figure 10-2. For five of the holes, survey of azimuth and dip could not be completed to the base of the holes due to hole collapse and compression. Survey of azimuth and dip also produced highly irregular results in two holes. The drillholes with unreliable survey data were not used to support geological modelling.

Table 10-2: 2016-2017 Delineation (REP) and Geotechnical (GT) Drilling by Lucara

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: SRK (2025)

10.2.2 2018-2019 Drilling: South Lobe 250-400 masl Definition

During 2018 and 2019, a total of 37 core holes were drilled for geotechnical and delineation purposes (Table 10-3) as part of the 2019 underground Feasibility Study. The drilling provided geological information below 400 masl in the South Lobe to support further resource development and additional geotechnical data for the kimberlite (KGR series). Drilling was also conducted to provide geotechnical information on the host rock stratigraphy (CR-GT series) and potential UG infrastructure locations (INFRA series). Four of the holes intersected the Centre and North Lobes between 700 and 500 masl (300 – 500 m below surface).

Drilling coverage from the 2016-2017 and 2018-2019 programs is shown in Figure 10-2.

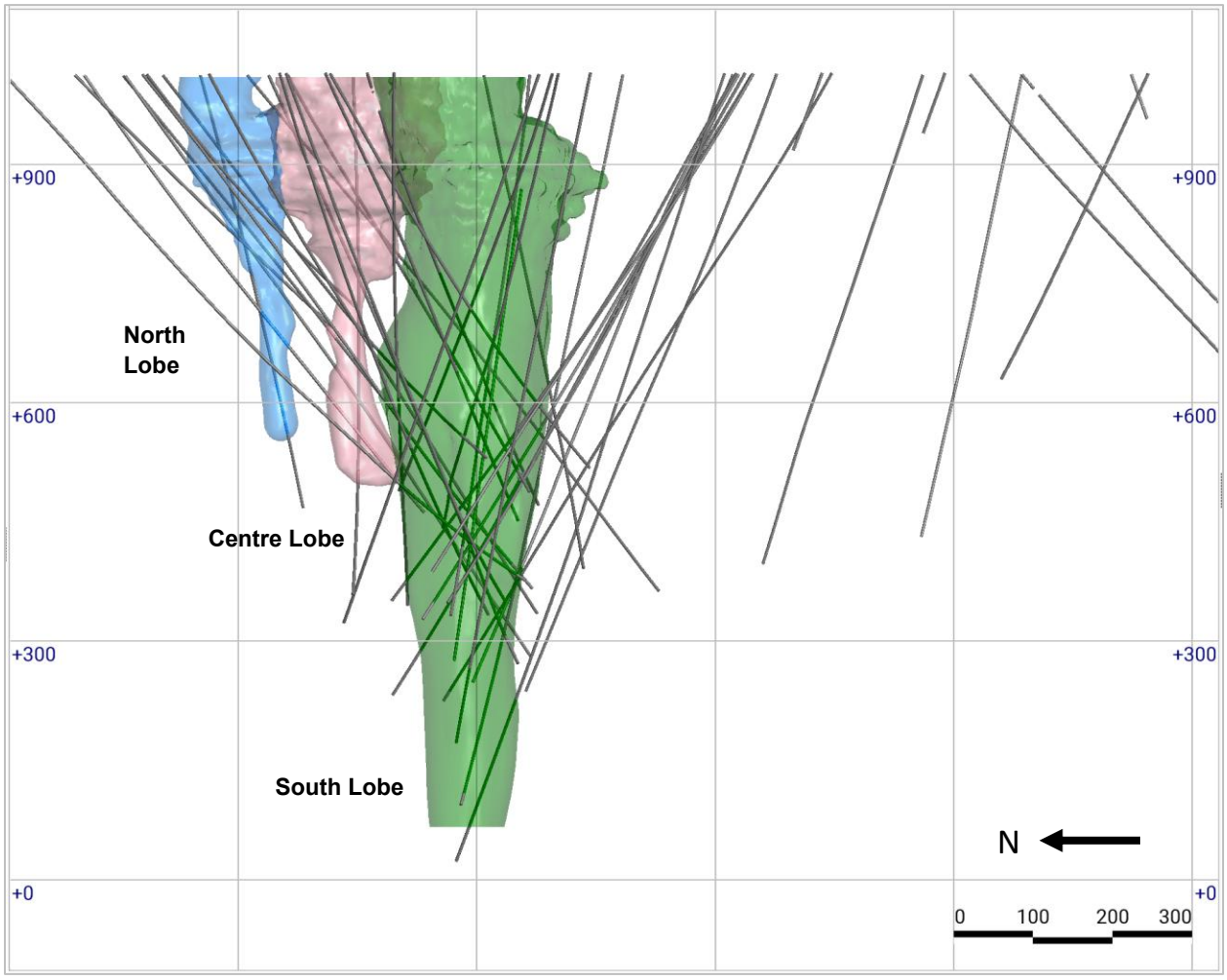
Table 10-3: 2018-2019 Delineation (KGR) and Geotechnical Drilling (CR-GT, INFRA) Drilling by Lucara

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

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

Source: SRK (2025)

Figure 10-2: 2016-2019 Delineation and Geotechnical Drilling by Lucara



Notes:
September 2025 geological model viewed looking east-northeast. Drillholes are shown as thin black traces.
Source: SRK (2025)

10.2.3 2020 Drilling: Shaft Geotechnical Investigation

A shaft geotechnical drilling program took place between May and December 2020. It consisted of two vertical diamond core holes designed to trace the proposed, and now sunk, UG mine shaft alignments along their entirety (Table 10-4). 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 ventilation shaft (V/S) and PS_GT_DD001 was collared at the approximate center of the proposed production

shaft (P/S). Both holes 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 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.

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				**1,514.5**		

Source: SRK (2025)

10.2.4 2025 Drilling: South Lobe

Five core holes totalling 1,781 m were drilled between July and September 2025 to provide additional geological and geotechnical information primarily for the minor kimberlite units at depth in the South Lobe and to inform placement of UG infrastructure (Table 10-5; Figure 10-3).

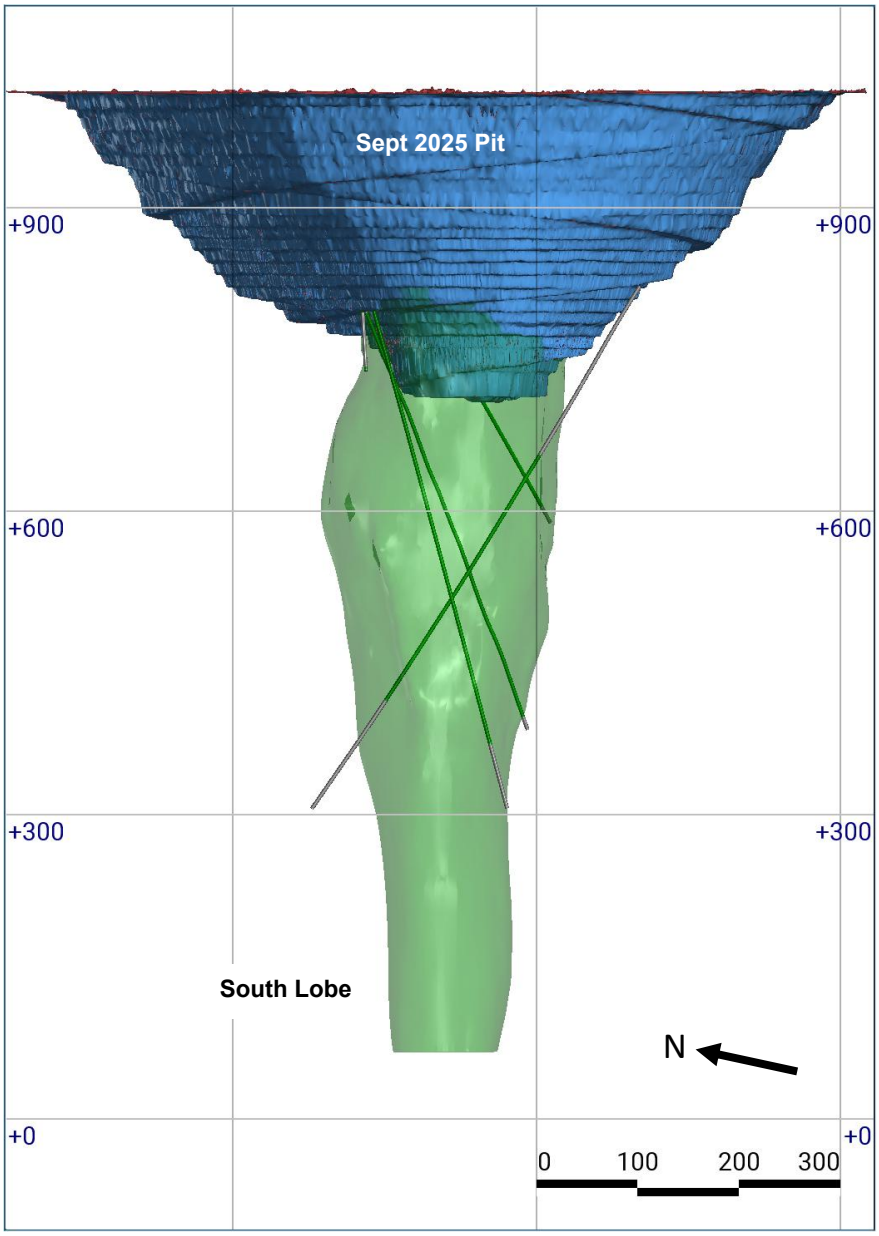
Drilling was conducted by Geomechanics Pty Ltd (GeoGroup) of Gauteng, South Africa using two Atlas Copco CS-1500 rigs and a Hanjin P7000 rig. Most of the core was HQ (63.5 mm diameter) with the upper 10-30 m of three holes being PQ (85 mm diameter). Drillhole collar locations were surveyed by the mine survey department. Downhole deviation was surveyed continuously using an industry standard, high accuracy north-seeking gyroscopic instrument.

Table 10-5: 2025 Delineation and Geotechnical Drilling

Drillhole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
2025-004-GGT	341482.0	7621741.4	798.90	445.75	119	-68
2025-010-GGT	341479.2	7621726.0	799.26	509.95	134	-75
2025-025-GGT	341484.7	7621749.0	798.53	60.53	199	-80
2025-027-GGT	341703.7	7621581.5	825.14	614.87	300	-58
2025-003-EM/PK(S)	341612.8	7621721.5	714.25	150.00	99	-57
Total				**1,781.10**		

Source: SRK (2025)

Figure 10-3: 2025 Delineation and Geotechnical Drilling



Notes:
September 2025 geological model and end-September 2025 pit. View looking northeast.
Source: SRK (2025)

10.3 Geological Logging Procedures

10.3.1 Historical Core Logging (2005-2007)

The pilot and delineation core holes were logged in the field in detail according to a standard operating procedure. The De Beers logs described each kimberlite unit, the nature of contacts between units and the kimberlite/country rock contacts, and also briefly described the country rock intersections (McGeorge et al., 2010). The historical holes were re-logged on site and/or reviewed in photographs for the 2018 geological model update (Section 7).

10.3.2 Core Logging 2016 to Present

Geological logging of drill core during Lucara's drilling programs was completed using standard operating procedures (SRK, 2019, 2025). Detailed kimberlite logging was completed by Lucara geologists under the guidance and supervision of Kimberley Webb, P.Geo. of SRK, who also logged or reviewed all core on site and/or in photographs.

A 'quick log' was generated for each hole at the rig site and geologists ensured the core was in order and that the core boxes and meterage were correctly labelled and marked. Total core recovery (TCR) and rock quality designation (RQD) data were collected at the rig. All core was photographed before being logged or sampled.

Detailed logs of the kimberlite recorded the key textural and component characteristics of the units including several specific to the kimberlite types found at KDM; these include olivine size and abundance, structure/fabric, the size, relative abundance and alteration/reaction state of mantle xenocrysts, and the abundance, size and relative proportions of different country rock xenoliths. Geologists used a binocular microscope during logging to aid in the identification of textures and components. Magnetic susceptibility readings were collected every metre downhole.

10.3.3 Mantle Xenocryst Counts

Systematic counts of mantle xenocrysts (garnet, ilmenite, orthopyroxene) were collected on drill core during Lucara's drilling programs to supplement core logging observations. The method provides semi-quantitative data to support characterization of kimberlite units and assessment of geological continuity within units. The procedure involves counting all mantle xenocrysts irrespective of alteration/reaction state along an arbitrary line drawn along the center of the core on a continuous basis downhole (2016-2017) or over 1 m intervals every 5 m downhole (2018-2019 and 2025).

10.3.4 Dilution Line Scans

Internal country rock dilution measurements by the line scan method were collected during all drilling programs to supplement visual estimates recorded during core logging.

The line scan procedure involves measuring all country rock xenoliths (CRX) 0.5 cm and larger along an arbitrary line drawn along the center of the core. For historical (2005 to 2007) drill core, the line scan measurements were conducted over 0.3 and 0.5 m intervals at ~4 to 5 m spacing downhole, and for the 2016-2017, 2018/2019 and 2025 drill core, by line scans over 1 m or 3 m intervals on a continuous basis downhole, with data collected per CRX type (basalt, basement, sedimentary). The minimum dilution percentage was calculated as: $Total\ CRX \geq 0.5\ cm$ / $Total\ Length$ of Core $Measured$ × 100.

10.4 Drill Core Sampling

Sampling of drillhole material in support of AK6 resource estimates is well documented in previous Technical Reports (McGeorge et al., 2010; Nowicki et al., 2018). This section summarises sampling of the 2018-2019 drill cores (Section 10.2.2) and is restated from Doerksen et al. (2024). Sampling of the five 2025 drillholes had not yet been conducted by the effective date of this Report.

A key requirement of the South Lobe Mineral Resource Estimate is demonstration of geological continuity within the M/PK(S) and EM/PK(S) with depth (Sections 7.4.3 and 14.4). Sample coverages achieved from the 2018-2019 drilling in the South Lobe are shown in Figure 10-4. Sampling was undertaken for bulk density, petrography and microdiamond analysis following standard operation procedures, as follows:

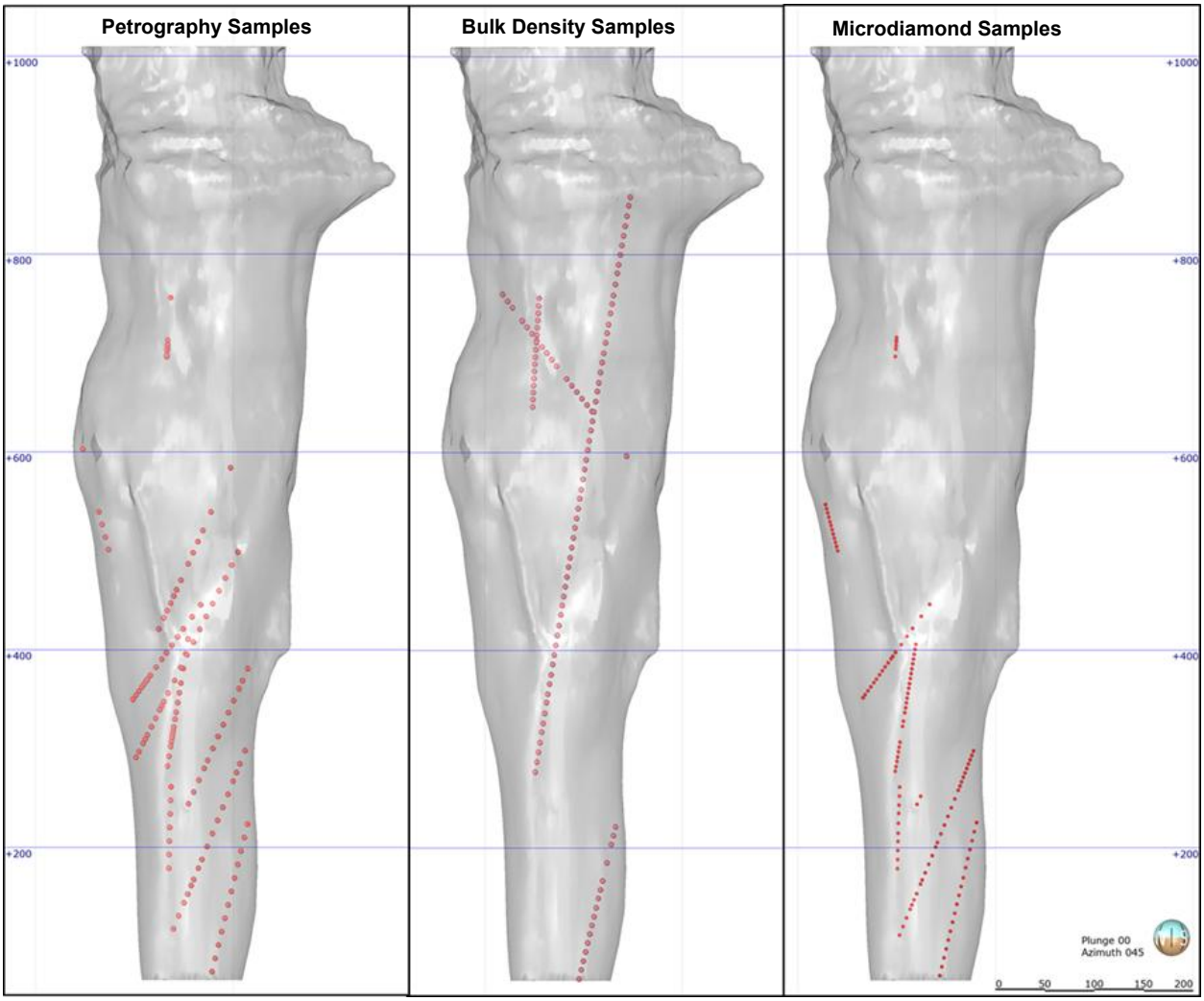
- Bulk density samples (n = 209, of which 188 are in the South Lobe). Samples each comprised 10 cm of whole core and were collected at regular 10 m intervals in six KGR / INFRA drill cores (four of which are in the South Lobe). It is noted that the historical and 2017 drill cores were comprehensively 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.

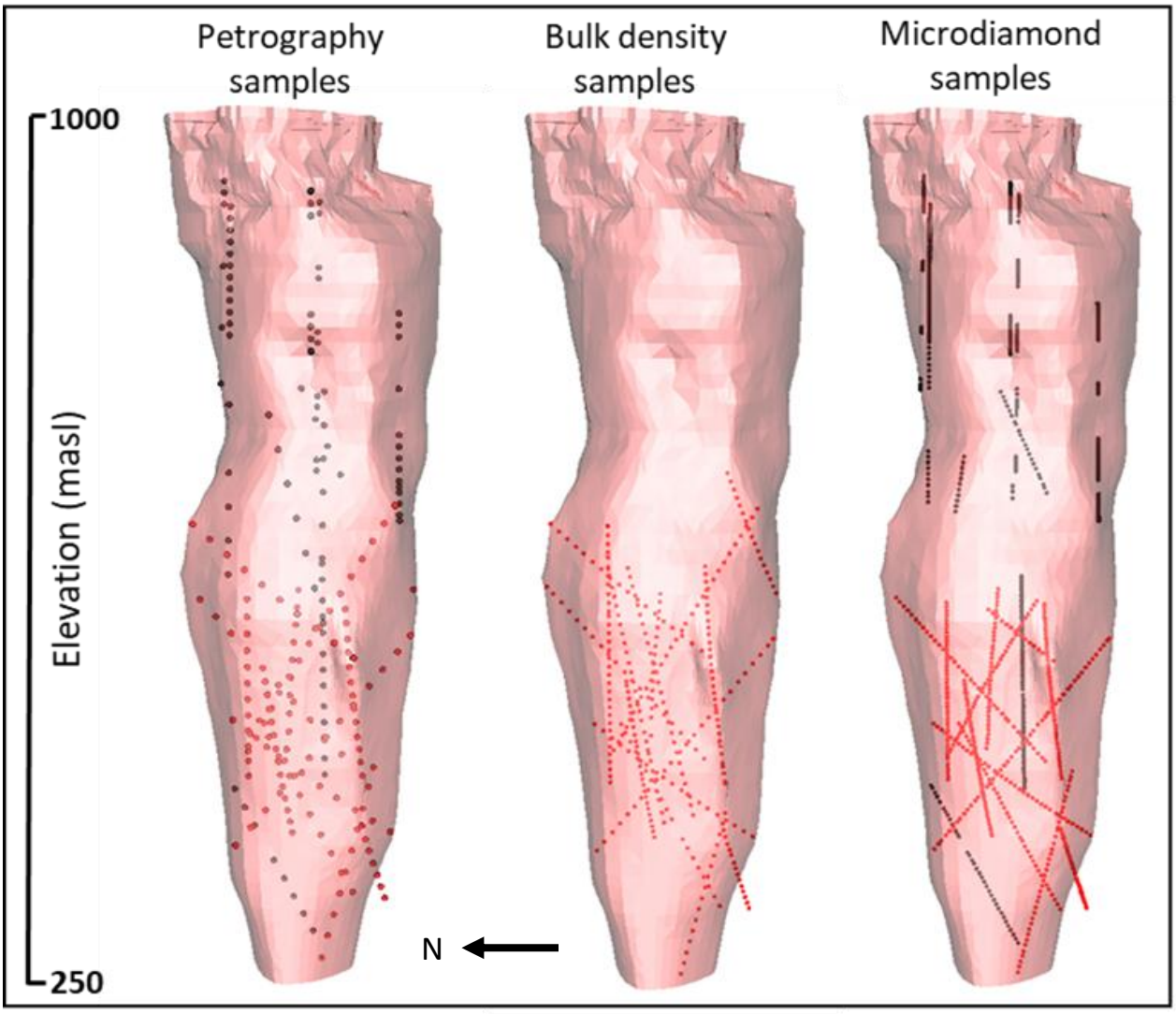
Figure 10-5 shows the locations of samples collected from 2016-2017 and historical drillholes (Nowicki et al., 2018).

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



Note:
Red = samples from 2018-2019 drillholes; view looking northeast.
Source: SRK (2019)

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



Note:
Black = samples from historical drillholes; Red = samples from 2016-2017 drillholes
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 2003-2007 are summarised in Section 11.1 and described in more detail in previous Technical Reports (McGeorge et al., 2010; Lynn et al., 2014; Oberholzer et al., 2017). Information relating to samples collected during Lucara's drilling programs in 2017 and 2018 (see Section 10.3) and which support this updated Mineral Resource Estimate is provided in Section 11.2.

11.1 Historical Samples (De Beers)

11.1.1 LDD Reverse Flood, 23" Drill Samples

These samples were collected by De Beers during Phase 1 and 2 exploration (Section 9.1) from the LDD holes described in Section 10.1. They formed the basis of the grade estimates in the upper portion of the South Lobe in the June 2023 MRE (Doerksen et al., 2024) as described in Section 14.4.1 of this Report.

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 in a secure cage. The drums were placed in sea containers with infrared motion detector surveillance and transported to the GEMDL in Johannesburg inside the sealed shipping containers 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 the GEMDL. The diamond recovery parameters were the same for both drilling phases. The GEMDL facility was fully ISO17025 certified at the time of sample processing. The recovery area is a security "red area" and is subject to access control, three tier surveillance and hands-off processing. The concentrates arrived 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 Lucara Samples: 2017 to 2024

All drill core petrography, bulk density and microdiamond sampling information in the current Technical Report pertains to previously-reported samples; no sampling of drill core has occurred since the 2018 core drilling campaign. Core holes from the 2025 drilling campaign had not been sampled by the effective date of this Report. Grab samples were collected from the OP in late 2024 for petrographic analysis to support internal domain contact delineation.

11.2.1 Petrography Samples

All petrography samples collected from delineation core holes in 2017 and 2018-2019 were labelled with the drillhole number, depth and way-up direction by Lucara Botswana geologists.

Grab samples collected from the OP in 2024 were labelled with a unique identifier and their locations were surveyed using a hand-held GPS. No further sample preparation was carried out on site.

Petrography samples were shipped to Vancouver Petrographics Ltd. (2017) and Precision Petrographics Ltd. (2019; 2024) in Vancouver Canada 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 measurements in 2017 and 2018-2019 were carried out on site by Lucara Botswana 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 and 2018-2019 are not of dry bulk density, and a minor adjustment to account for internal moisture content and to ensure compatibility between the Lucara and historical datasets was applied as documented in Section 12.

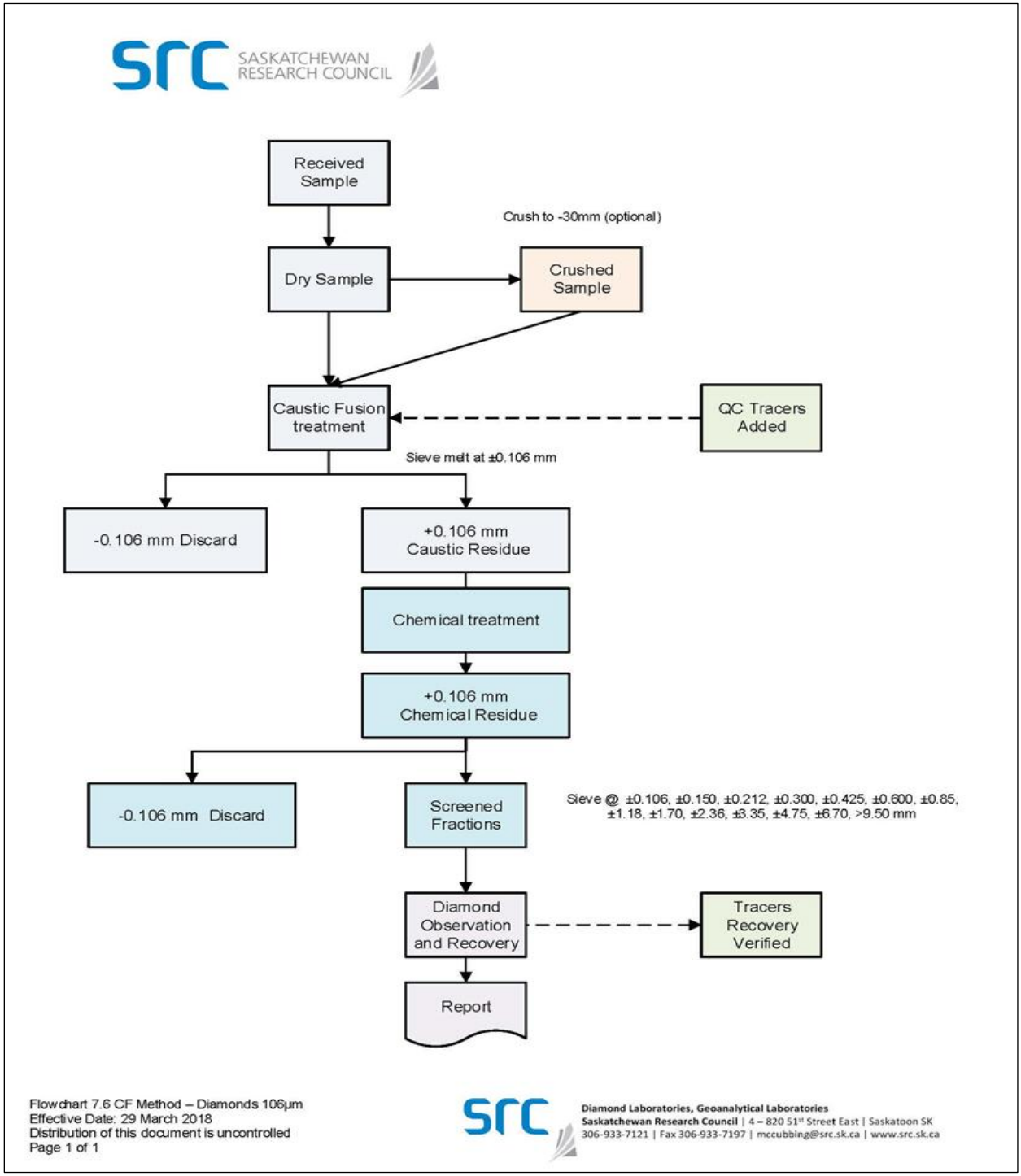
11.4 Microdiamond Samples

No preparation of microdiamond samples collected in 2017 and 2018-2019 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 from the time the samples were processed 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. Microdiamonds 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.

In the QP's opinion, the sample preparation, security and analytical procedures meet or exceed industry standard practices, and the QA/QC results demonstrate acceptable levels of accuracy.

Figure 11-1: Processing Flowsheet for Microdiamond Samples Processed at the SRC



Source: SRC (2019)

12 DATA VERIFICATION

12.1 Geological Model

12.1.1 Drillhole Collar and Orientation Surveys

Historical (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 2019 geological model update (Doerksen et al., 2024). The 2025 drillholes were surveyed using a gyroscope-based system. Ms. Webb reviewed the recent survey 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 reviewed all 2018/2019 and 2025 drill cores intersecting the South Lobe on site or in photos to verify the mine-generated drill logs and data supporting geological continuity within domains. Logged contacts were verified 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 country rock xenoliths greater than 0.5 cm in size were determined for historical (2005-2007) drill core by line scan measurements over 0.3 and 0.5 m intervals at ~4 to 5 m spacing downhole, and for 2016-2017, 2018-2019, and 2025 drill core by line scan over 1 m or 3 m intervals on a continuous basis downhole. The methods are considered by Ms. Webb to be appropriate and consistent with industry standard 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 delineation and pilot drill cores and Lucara's 2017-2019 delineation and geotechnical drill cores. Dr. Grütter considers the methods used to be in line with industry standard practice. He reviewed the bulk density database and verified that samples were correctly coded according to domains in the current geological model. No significant issues or discrepancies were found.

12.2.2 Microdiamond Data

Microdiamond assay results for samples from drill core were compiled from original laboratory 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; recoveries of these synthetic diamonds are reported along with microdiamond recovery results. Dr. Grütter reviewed the microdiamond sample and quality control results, and no significant issues were noted.

12.2.3 2006-2007 Macrodiamond Data

Large-diameter drilling and bulk sampling campaigns conducted in 2006 and 2007 provided the primary macrodiamond data utilised in prior mineral resource estimates. These data were found to be consistent with the original bulk sampling documentation by Mr. Revering, an independent resource QP (Section 12.2.3 of Doerksen et al., 2024). Dr. Grütter examined digital records of the 2006-2007 macrodiamond data and retraced their utilization in the June 2023 Mineral Resource Estimate. No significant issues or discrepancies were found.

12.2.4 Run-of-Mine Macrodiamond Data

Selections of discrete daily ROM production over the period January 2016 to September 2025 constitute the primary macrodiamond data utilised in support of the current Mineral Resource Estimate. Dr. Grütter reviewed compilations made by Lucara staff of daily production provenance, weightometer records, select process plant metrics, sieve size data for macrodiamonds recovered, and also reviewed the consolidation of select, discrete daily records by calendar month. No significant issues were noted.

12.2.5 Production and Sales Data

Annual sales data dating back to the start of mining operations in 2012 were provided to SRK as part of the 2025 Mineral Resource update. Dr. Grütter reviewed these data in the context of general reconciliation of past production and diamond revenues with the current Mineral Resource Estimate. Explicit tracing by Lucara staff of discrete daily diamond production through

consolidation into parcels for final sale was also reviewed by Dr. Grütter, in the context of constraining characteristic diamond revenues for major geological units in the KDM resource. No material issues or discrepancies were noted.

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

12.3 Mineral Reserve Estimate

Multiple site visits were conducted by the QP during the construction of the UGP.

Mineral reserve estimates were based on the 2025 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 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%.

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. Key inputs and assumptions used in cost estimates were verified by reviewing invoices, contracts and sales data. Operating costs were verified against company financials and annual reports.

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

12.4 Mineral Processing and Metallurgical Testing

First Inspection Report by Tomra included drill core samples collected and testwork supervised by site representatives. The location of the samples tested was not identified. Further test runs at site through the crushing/sorter circuit using drill core were completed and the results indicated the sorters can identify waste material from diamonds.

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 comminution data 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

On site visits were conducted by Principal Engineers and QPs from SRK (2019-2023), Itasca Consulting Group (2019-2024), RHDHV (2022), Beck Engineering (2024-2025) and Middindi Consulting (2025). The most recent QP visit (Middindi,2025) and associated work was aimed, inter alia, at:

- Inspecting ambient rock mass conditions in the open pit;
- Conducting validatory drill core logging;
- Consolidating the extensive quantity of geotechnical data that informs country rock and kimberlite rock mass quality;
- Reorganizing and packaging of strength test data;
- Characterisation of the host and ore zone rock mass;
- Empirical assessments of free and assisted cave zoning within the pipe;
- Provision of rock properties for numerical assessments;
- Review and revision of the ground control management plan (GCMP); and
- Assistance with drafting of the instrumentation and monitoring strategy.

The initial (2019-2023) QAQC of core logging, wireline logging, field and laboratory testing and sampling activities was carried out by SRK Consulting (South Africa) following standard operating procedures implemented by SRK. An additional QAQC of the historical database was conducted by the QP in 2025 the results of which are contained in document reference: *Middindi (2025). Geotechnical Database Consolidation and Design Validation, Karowe Diamond Mine, Botswana*.

Several drilling campaigns were concluded to obtain geological and geotechnical data to inform the current mine design. This includes the 2017-2018 drilling campaign, the 2019 geotechnical investigation program and the recent 2025 infill campaign. All drilling programmes provided characterisation for both the country rock and South Lobe kimberlite. Geotechnical interpretations take cognisance of the updated 2025 geological model and geotechnical data up to the 2019 programme. The number of boreholes, meterage, field and lab tests that supports the country rock and Kimberlite geotechnical characterisation is extensive and listed in Table 12-1.

Table 12-1: Geotechnical data acquisition summary

Geotechnical Data	Country Rock	Kimberlite	Study Campaign
Number of boreholes	52	29 of 52	2017-2019
Meterage (m)	36,236	6,668	2017-2019
Validatory logging (BHs)	9	9	2025
Validatory meterage (m)	4,636	1,490	2025
Contact alteration (BHs)	25 (48 pierce points)		2025
Micro structural data (BHs)	19	19	2019
Structural pole count	19,514	7,182	2025
Rock tests	2,855	331	2019
Field PLT tests	6,662	1,216	2019

Source: Middindi (2025)

The drilling program included completion of 36,236 m of geotechnical drilling from 52 drillholes through both country rock and orebody to support 7,878 field strength (point load) tests and collectively 3,186 valid laboratory tests. The Total Level of Data Confidence (TLDC) was quantified (SRK, 2019) specifically for the laboratory testing specimens and indicates that most tests met the minimum criteria for the upper limit of the feasibility level study of between 60 - 75%. Data gaps have been listed and incorporated in the risk register with recommended mitigating measures.

12.6 Water Management and Hydrogeology

KDM is a brownfields site with over 10 years of actual mine dewatering data available (2012-2025) on which the aquifer system behavior and pressure response could be analysed and used in the model calibration. The level of data gathered and analysed is beyond feasibility study requirements with 23 pumping tests, 58 packer tests and 400 hydrochemistry tests. During the last 13 years of mining operation, there are measured groundwater levels that have been obtained from over 40 monitoring locations and measured pumping rates from more than 25 dewatering boreholes. The inflow rates from shaft sinking and from underground developed were measured. These operational data, which are analogous to the field scale hydraulic testing, were used in the model update and calibration. Existing data was reviewed and analysed 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 for FS requirements.

12.7 Mining Methods

Multiple site visits were conducted by the QP since 2020, with the most recent on Nov 23, 2024, 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 planned and established mine development locations including shafts, raises, and lateral drives;
- 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;
- Inspection of supplier facilities from which mine infrastructure has been or will be sourced, including ventilation, pumping, and mobile equipment;
- Engagement with key development contractors including shaft and lateral development to qualify costs and timelines; and
- Construction oversite of shafts and shaft sinking infrastructure to qualify functional criteria.

Mineral resource models were imported to mine software and queried for tonnes and grade to validate resource tables stated herein, and cave simulation results were equally imported to mine software to validate tonnes and grade of production schedules and ensure mine design and schedule was properly honoured. Site visits, photos of development, and survey scans have informed the status of mine excavations as at the effective date.

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

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 2024 sustainability report, as cited in the section "Environmental Studies" and "References".

The information provided in this Report is provided without limitations. The QPs have over 25 years of relevant experience. Their most recent visits to KDM since the 2023 FS were Feb 13-17, 2023 and Nov 21-25, 2024. Based upon this, the QPs are 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 2019 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 QP's opinion that the data utilised 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 witnessed site power supply upgrades made between 2020 and 2023 which includes:

- New 132 kV 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 LOM 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 LOM requirements.

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 FRDs 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 LOM, 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 capital and operating cost estimates for OP mining, processing, tailings, G&A, cost of sales, corporate costs, taxes, and mine closure. Lucara Botswana costs have been validated against annual financial reports.

All other UGP capital and operating costs have been estimated or managed by the QP through a combination of historical costs, current commitments and contracts, and first principles estimates.

Multiple bids have been used to validate key consumable, equipment, and contractor costs. Purchase orders, contracts, invoices, and project specific proposals were reviewed to verify cost inputs. 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 an AACE Class 3 Estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineral Processing Testwork

The KDM processing plant has been treating unweathered South Lobe ore since 2015 from the open pit. In support of the FS tests were completed to verify the compatibility of the ore at depth with respect to processing the material in the existing comminution circuit.

A comminution test program was conducted to investigate the characteristics of the South Lobe material below the OP to determine if the mill is suitable for deeper EM/PK(S) ore. The testing was completed at Base Metallurgical Laboratories in Kamloops, B.C., Canada.

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 shales 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. First Inspection Report was conducted by Tomra at their testing facilities in Germany.

13.2 XRT Testwork

The preliminary XRT (X-ray Transmission Technology) testwork conducted in May 2019 "First Inspection Report for the Lucara Underground Diamond Study" by Tomra Sorting Solutions (Tomra) in Germany and subsequent confirmatory testing in 2025 "Carbonaceous Shale Recoverability by XRT Sorters Testwork" conducted by Lucara mine site personnel, demonstrate that the COM (Common Belt) Tertiary XRT sorter can distinguish liberated diamonds from the principal host rock lithologies found within the Karowe underground deposit. The original Tomra inspection indicates that all tested waste lithologies—including sandstone, granite, and carbonaceous mudstone—were correctly recognised as high-density material under the standard sorting algorithm (Setting 1). However, when the sensitivity was increased (Setting 2), the carbonaceous mudstone exhibited a response similar to diamond, resulting in misclassifications. These findings established the operational sensitivity boundaries for XRT sorting in carbon-rich lithologies.

Drill core samples of carbonaceous shale were collected by site in 2025 and processed through the crushing/sorting circuit. The testwork focused on the recoverability of carbonaceous shale using both the LDR (Large Diamond Recovery) and Middles XRT sorters. The Carbonaceous Shale Recoverability by SRT Sorters Testwork, Karowe Diamond Mine, 2025 indicated generally low recovery yields from the LDR sorter (2.5 – 4.5%), with one anomalous test yielding 24.2%. Conversely, yields at the Middles 1 (+8 mm - 19 mm) sorter ranged consistently between 16 – 18%. The elevated yield observed in finer fractions is attributed to the material's low density, brittleness, and propensity to fracture during handling, which leads to increased liberation of smaller fragments. It was also observed that the sorter's automated ejection valve test cycles may contribute to inflated concentrate yields during processing of fine fractions. The LDR-sorted

material, being coarser, exhibited less misclassification but lower overall recovery due to reduced liberation and higher particle stability.

These collective results confirm that the carbonaceous shale material is light, brittle, and potentially responsive to XRT sorting under standard configurations, though optimisation is required to reduce carryover and anomalous ejection events. The carbonaceous mudstone remains distinguishable from kimberlite and diamond under the correct parameterisation (Setting 1), supporting the broader conclusion that the mineralisation is amenable to sensor-based sorting.

It was also noted that the 2019 testing preceded the implementation of the Tomra OBTAIN software platform, which is expected to mitigate the sensitivity and inconclusiveness issues identified in earlier analyses. Even under fully optimised conditions, misplacement of carbonaceous material into the concentrate stream should be anticipated and incorporated into downstream plant design and recovery modelling.

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 and Figure 13-1.

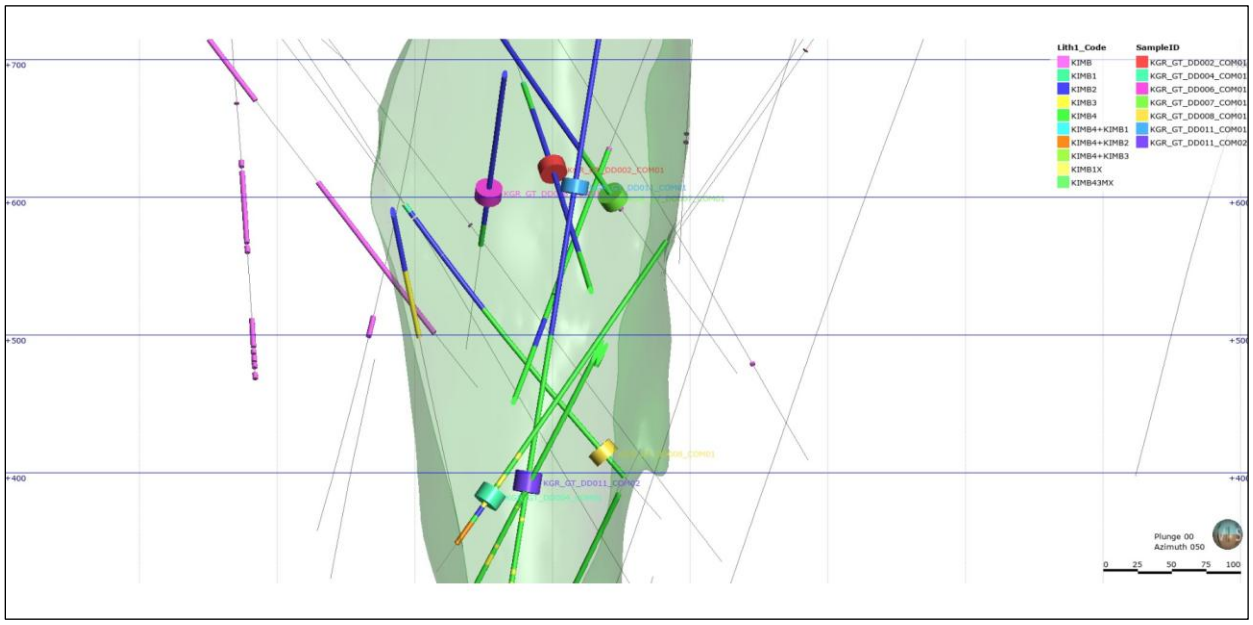
Table 13-1: Comminution Testwork Sample Selection

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

Sample ID	Hole ID	From (m)	To (m)	Lithology*	From	Mass (kg)
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)

Figure 13-1: 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 impact breakage (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 microns (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/m³	%	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

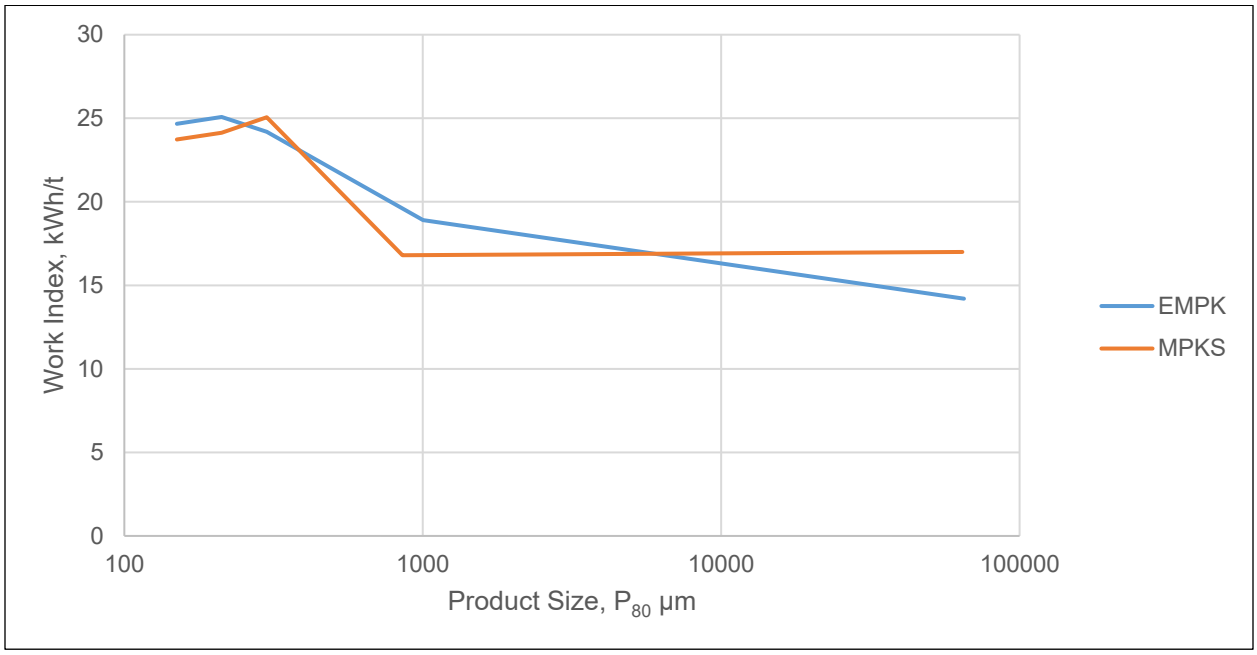
Source: BaseMet (2019)

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 samples. The samples tested are amenable to milling in the existing AG process plant.

Figure 13-2 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 13-2: 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.

The QP, Kelly McLeod, P.Eng., acknowledges that there are numerous factors that may influence the potential for economic extraction. At present, aside from those already addressed in the Mineral Processing and Metallurgical Testing sections of this report relating to comminution and sorting testwork, no additional specific processing factors or deleterious elements have been identified as having a significant effect.

13.4 Processing Assumptions

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

The KDM processing plant is expected to continue to achieve an annual throughput of 2.85 Mt of feed once the underground material is mined.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The KDM has been in operation since 2012, and as of the end of September 2025 the mined OP extends to a depth of approximately 715 masl (300 m below surface). The September 2025 Mineral Resource update for the KDM is underpinned by the data, estimation methods and results of the June 2023 Mineral Resource Estimate or MRE (Doerksen et al., 2024), with the following updates:

- Relegation of the Centre and North Lobes to an Unclassified status as a result of this material becoming stranded in the north wall of the KDM OP and requiring (uneconomic) pit wall pushback and haul road reconfiguration to be accessed. Using 5% of monthly headfeed tonnes as a threshold, the most recent consequential production from the North Lobe occurred during February 2023, and during February 2025 for the Centre Lobe. The Unclassified status extends to known depths for Centre and North Lobe material of approximately 500-550 masl (450-500 m below surface) since current underground mine plans are focused on extraction of South Lobe resources using major infrastructure locations at 310-340 masl that are considered unsuitable for (economic) underground access to the Centre and North Lobes;
- As-built survey of the OP mine as of September 30, 2025;
- In-pit mapping data of internal and external kimberlite contacts for the South Lobe;
- Limited diamond core drilling conducted during Q3 2025, and primarily concerned with the distribution of six minor geological units occurring at depth in the South Lobe;
- An updated geological model for the South Lobe incorporating in-pit mapping since Q3 2019 and the Q3 2025 drilling data;
- Incorporation of pre-existing density and 2017-2019 microdiamond sampling data for six minor geological units occurring in the South Lobe predominantly at depth;
- Reconciliation of the June 2023 MRE with actual recovered diamond grades and size frequency distributions (SFD) for in-pit depleted portions of major geological units M/PK(S) and EM/PK(S) in the South Lobe, based on daily production data to September 30, 2025; and
- Revised diamond pricing based on complete sales data to December 31, 2024 and partial sales data to September 30, 2025

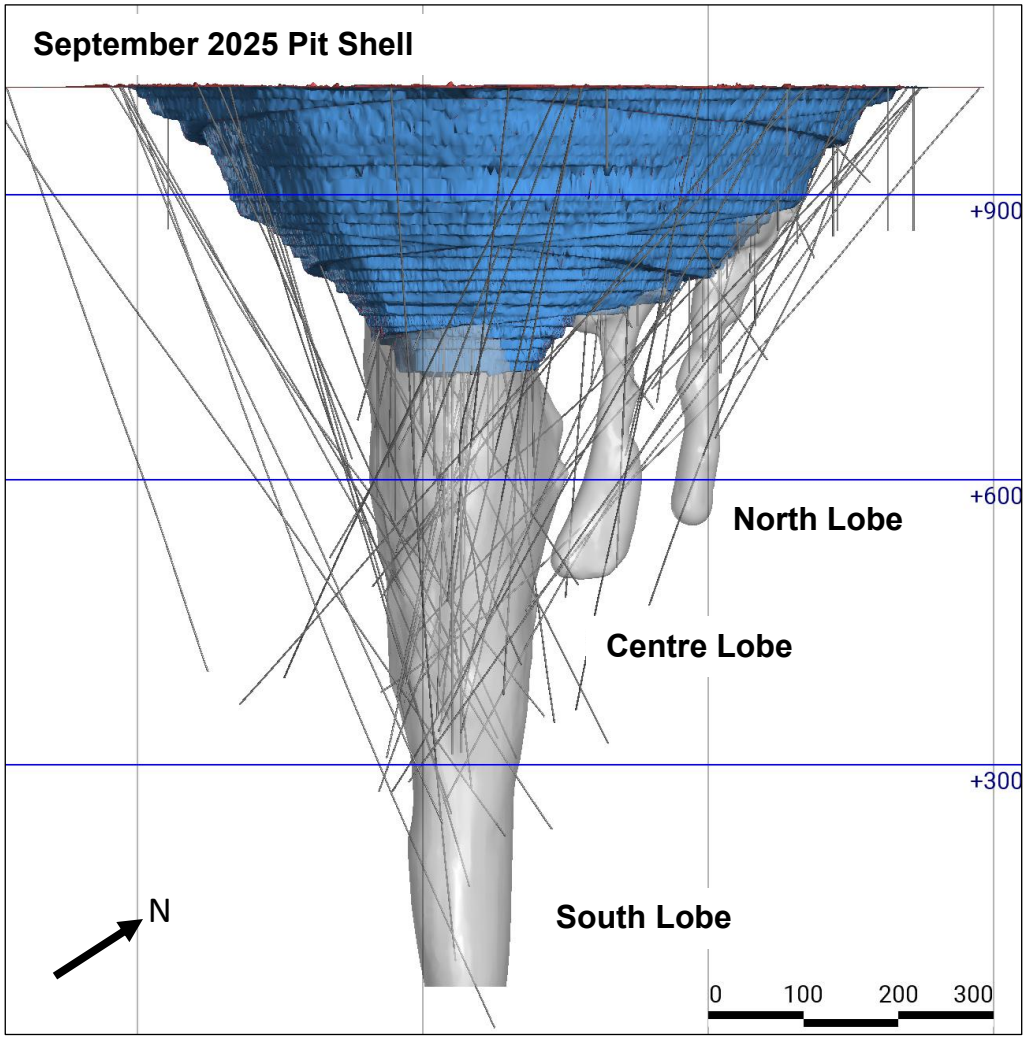
The terms microdiamond and macrodiamond in the context of this Report are defined as follows:

- Microdiamonds: are typically smaller than 1.00 mm and have been recovered from kimberlite drill core using caustic fusion, at a bottom screen size of 105 µm (0.105 mm); and

- Macrodiamonds: are 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 September 30, 2025, and all drilling used to support the MRE.

Figure 14-1: Geological Model of the KDM Kimberlite



Note:
Kimberlite (grey), the September 30, 2025 mined OP, and all drillhole traces. View looking northwest.
Source: SRK (2025)

The current geological model and MRE were generated in Seequent's Leapfrog Geo and Edge modelling software. The block model is comprised of a whole-block format using the following configuration parameters:

- Block model X, Y, Z origin of 340950, 7621200, 1078, respectively, with no rotation;
- Block size of 3 x 3 x 3 m; and
- Model extents (by # of blocks) of 416, 368 and 348 along the X, Y, Z axes.

The block model contains estimates of volume and global estimates of density and tonnes for all geological domains internal to the South Lobe, paired with global estimates of diamond grade. Further details of the estimation methodology are provided in the following sections.

14.2 Resource Domains and Volumes

The geological model for KDM is described in Section 7.4 of this Report and volume estimates of the unmined, in situ internal kimberlite domains are listed in Table 14-1. All kimberlite domains that have been mined as of September 30, 2025, are excluded from the volume estimates provided in Table 14-1.

Table 14-1: In Situ Volumes of Unmined Kimberlite Domains in South Lobe as of September 30, 2025

Kimberlite Domain	Volume (Million m^3)	Volume (% of total)
South_M/PK(S)	5.39	39.7
South_EM/PK(S)	7.35	54.2
South_KIMB3	0.45	3.3
South_KIMB1	0.07	0.5
South_KIMB4a	0.10	0.8
South_KIMB5	0.06	0.4
South_KIMB6	0.00	0.0
South_WM/PK(S)	0.13	0.9
RFW-ALT	0.02	0.1
Total	**13.57**	**100**

Note:
Figures have been rounded to reflect the accuracy of the estimate.
Source: SRK (2025)

14.3 Bulk Density

A total of 2,127 dry bulk density measurements were used for the September 2025 Mineral Resource update. Bulk density values were assigned to the block model based on two criteria: (1) density variation with depth where applicable, and (2) statistically representative values (mean or median) per resource domain (Table 14-2). Measurements from the weathered zone (above approximately 950 masl; mined out) were excluded from the statistical analysis.

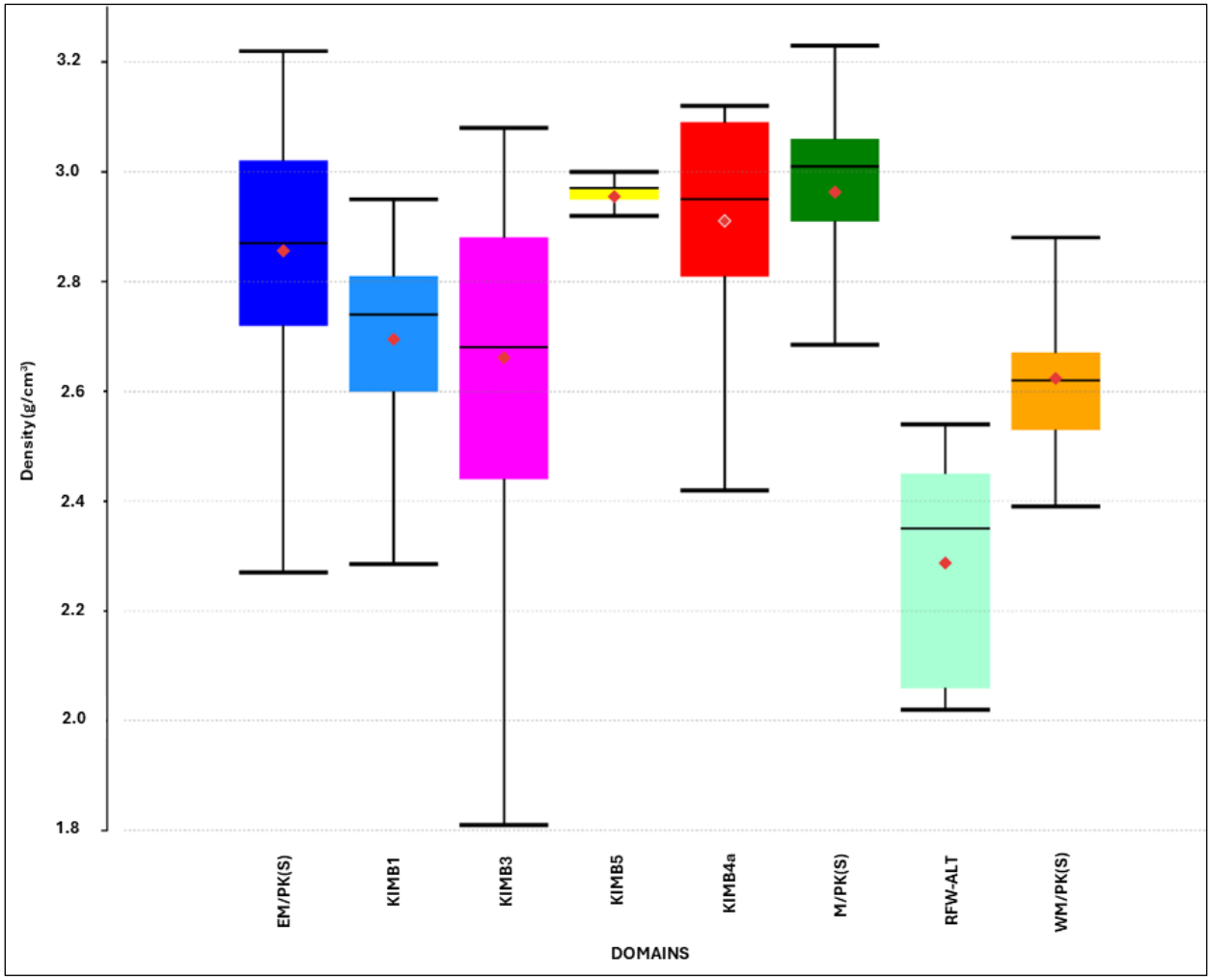
Table 14-2: Dry Bulk Density Sample Statistics for KDM Kimberlite Domains and values applied in the Block Model

Kimberlite Domain	Sample	Mean	Standard Deviation	Median	Block Model Applied Density	Rationale
	Count	(g/ cm^3)	(g/ cm^3)	(g/ cm^3)	(g/ cm^3)	
M/PK(S)	1117	2.96	0.15	3.01	2.97	Mean
EM/PK(S) >= 560 masl	244	2.73	0.18	2.92	2.73	Mean
EM/PK(S) < 560 masl	265	3.00	0.18	2.92	3.00	Mean
KIMB1	35	2.69	0.16	2.74	2.74	Median
KIMB3	42	2.66	0.28	2.68	2.68	Median
KIMB4A	15	2.91	0.21	2.95	2.95	Median
KIMB5	5	2.95	0.05	2.97	2.96	Mean/Median
KIMB6	n/a				2.96	No samples in KIMB6, therefore used KIMB5 BD value
WM/PK(S)	53	2.62	0.14	2.62	2.62	Mean/Median
RFW-ALT	11	2.29	0.19	2.35	2.35	Median

Source: SRK (2025)

A total of 2,127 density measurements from the South Lobe were evaluated by resource domain. With the exception of EM/PK(S), all domains either contained insufficient samples for a depth analysis or exhibited no clear trend with elevation. Box-and-whisker plots for each resource domain informed the selection of density values assigned to the block model (Figure 14-2).

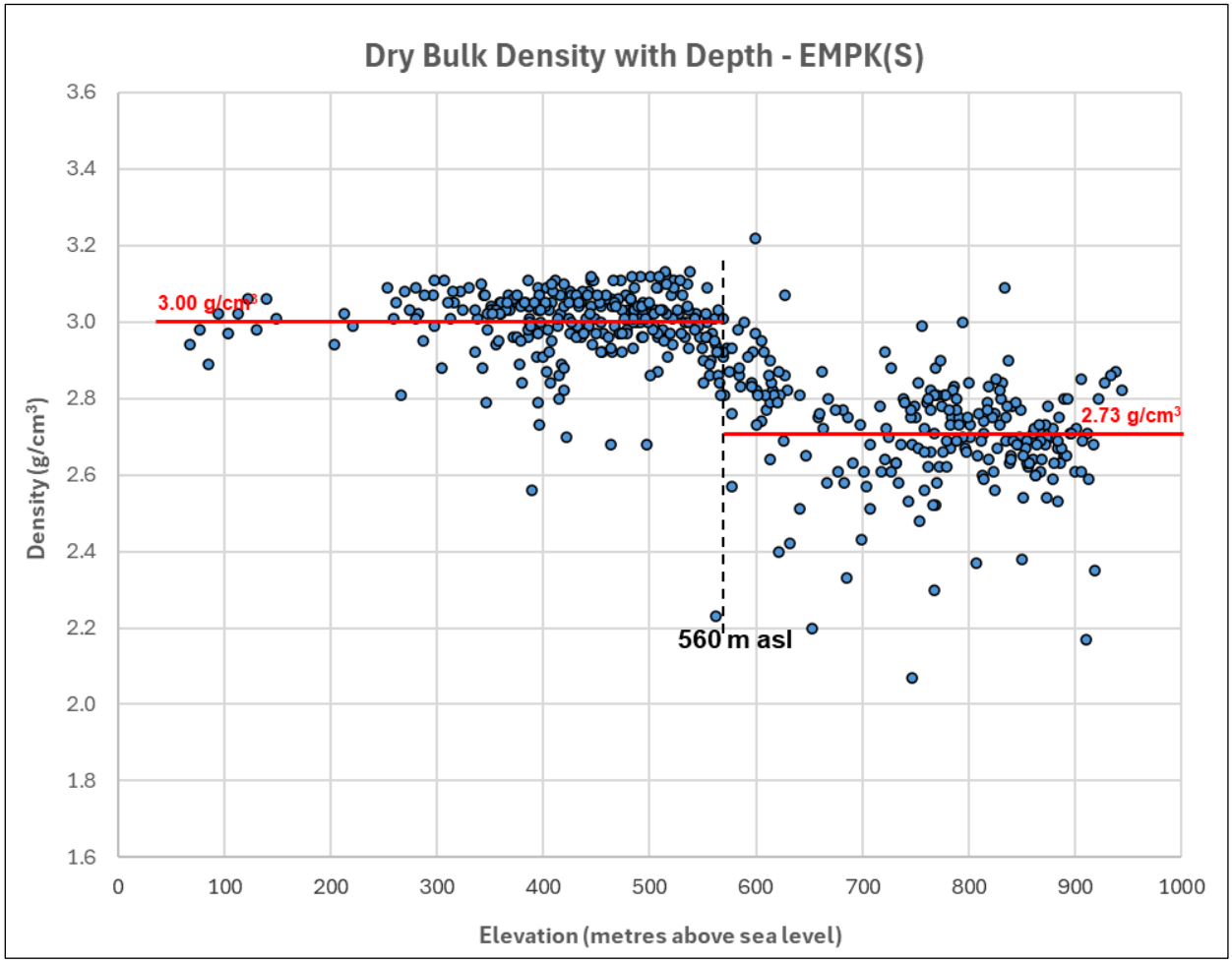
Figure 14-2: Box-and-Whisker Plot of Dry Density Values per Resource Domain in the South Lobe



Note:
Thresholds shown correspond to minimum, first quartile, median, third quartile, maximum and mean (red diamond).
Source: SRK (2025)

The EM/PK(S) domain exhibits a distinct density transition at approximately 560 masl (450 m below surface) (Figure 14-3). This elevation 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 likely due to the EM/PK(S) being constrained to a narrow zone along the eastern side of the South Lobe above 450 m depth (Section 7). The block model for this domain was assigned densities in two zones: 2.73 g/cm³ above 560 masl and 3.00 g/cm³ below 560 masl.

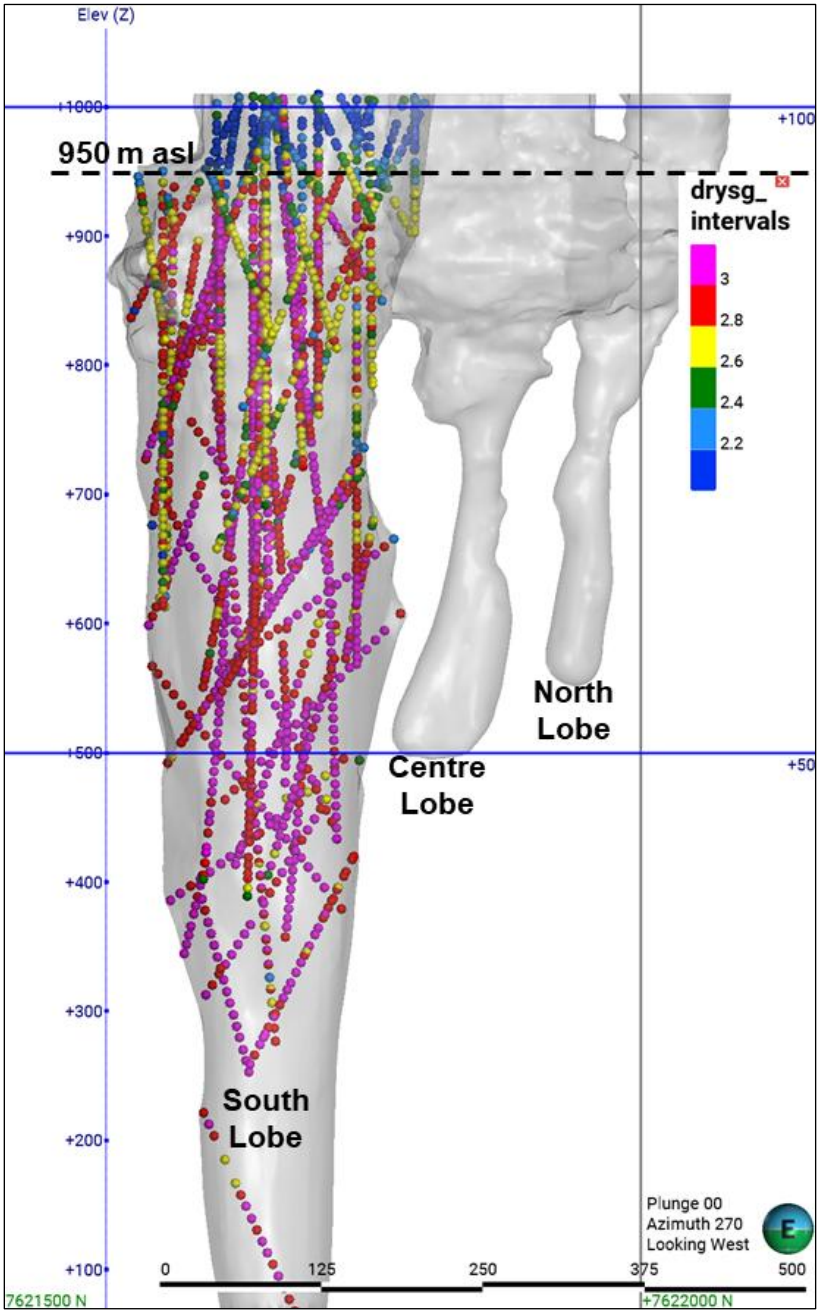
Figure 14-3: Dry Bulk Density with Depth – EM/PK(S)



Source: SRK (2025)

Figure 14-4 shows the dry density sample locations in the South Lobe. The 950 masl elevation approximately corresponds to the lower boundary of the upper calcretized and weathered kimberlite and country rock breccia zone. Average dry density values in this zone are significantly lower than density values of the kimberlite below and were not used in the density analysis per domain.

Figure 14-4: Drill Core Dry Bulk Density Sample Location Map



Note:

Dry density units of g/m^3. Black dashed line at 950 masl indicates the approximate base of the upper weathered/breccia zone reflected in generally lower densities. These samples are shown in the figure but were not used in the density data analysis. The depth of the September 2025 OP in the South Lobe is 715 masl).

Source: SRK (2025)

14.4 South Lobe Grade Estimation

14.4.1 Grade Estimation for the 2023 MRE

Two distinct methodologies were used by independent QP Mr. Revering to estimate diamond grades for the South Lobe during development of the June 2023 MRE (Doerksen et al. 2024):

- Local estimation of block grades shallower than approximately 570 masl were based on 2006-2007 large diameter drillhole (LDDH) bulk sample data; and
- Global estimation of diamond grades deeper than 570 masl were based on the correlation of microdiamond abundance with macrodiamond grade obtained from LDDH bulk sampling.

Section 14.3 of the 2023 FS report (Doerksen et al. 2024) provides details of the macrodiamond and microdiamond data used, grade capping, local grade estimation, global grade estimation and an adjustment made for process plant recovery efficiency at +1.25 mm bottom cut-off. Mr. Revering summarised this work as illustrated in Figure 14-5 and as follows:

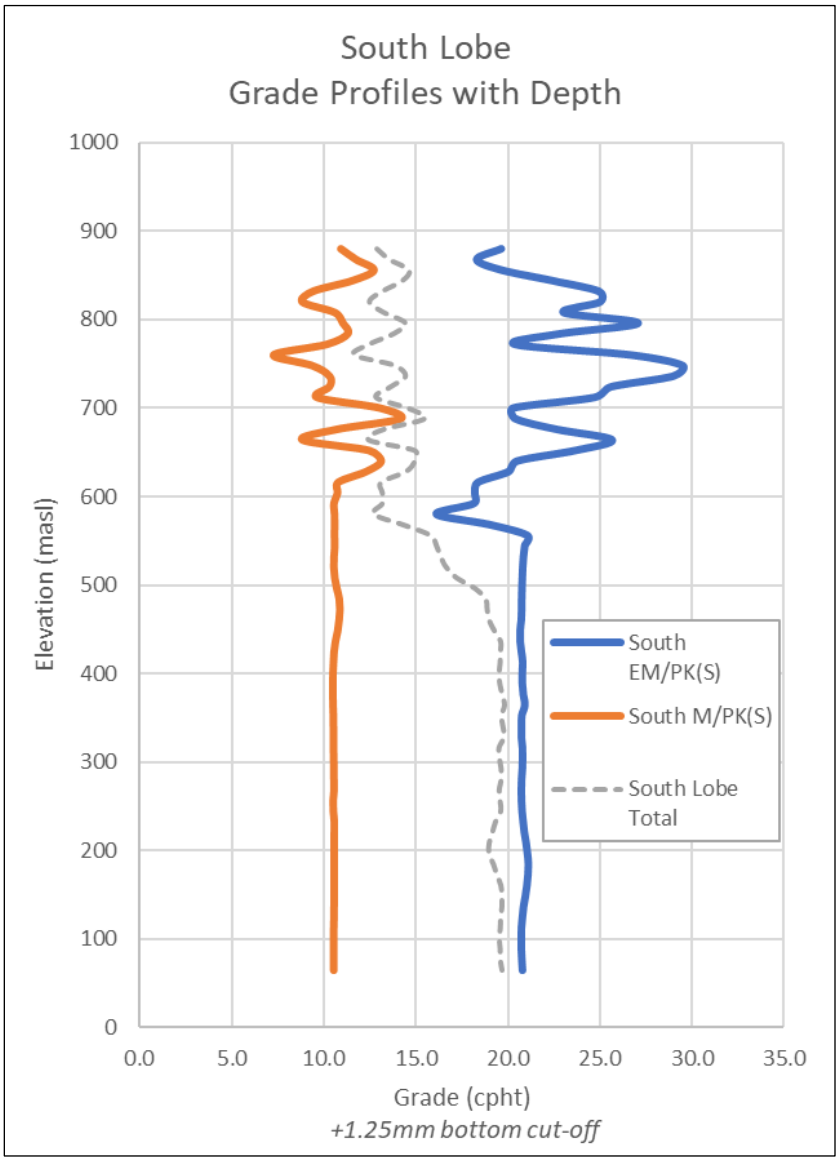
> *"Vertical profiles of recoverable grade (cpht) at a bottom cut-off of +1.25 mm for the South Lobe are provided in Figure 14-5. The profiles represent the grade estimation approach adopted for the [June 2023] Mineral Resource Estimate and reflect variable local grade estimates supported by LDDH bulk 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 5 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 14-5 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."*

Dr. Grütter comments that, instead of repeating the above-mentioned grade estimation procedures, an independent reconciliation of +1.25 mm diamonds recovered from mining of South Lobe kimberlite would serve as a timely and appropriate check on the accuracy of the June 2023 grade estimates, in particular since:

- No bulk sampling to depth has occurred since the 2006-2007 LDDH campaign;
- No microdiamond sampling has occurred since the 2019 core drilling campaign;
- Mining of the KDM OP is expected to cease at the end of 2025 and full-scale underground production of only South Lobe kimberlite is anticipated in the first half of 2028 (Lucara, 2025a); and

- Daily Mine Call Factors for South Lobe material mined from the OP appear to improve when local grade estimates in the June 2023 block model are appropriately substituted with global grade estimates for M/PK(S) and EM/PK(S). This observation suggests that estimates of locally variable grade underpinned by 2006-2007 LDDH bulk sample data may be poorly reproduced at the scale of resource extraction and supports independent reconciliation of June 2023 grade estimates with grades recovered for already-depleted kimberlite from the South Lobe.

Figure 14-5: Recoverable Grade Profile with Depth for the Dominant South Lobe Domains



Source: SRK (2023)

14.4.2 2025 Grade Reconciliation

Section 12.2.4 of this Report outlines verification of the run-of-mine data used to check and reconcile grades recovered from South Lobe kimberlite: selections were made of discrete daily plant feed consisting of either 100% M/PK(S) or 100% EM/PK(S) kimberlite, with the resultant daily data consolidated per calendar month. The available dataset contains 100% M/PK(S) entries for 96 of the 117 months in the period January 2016 to September 2025, while 100% EM/PK(S) entries occur for 29 of the 117 months. Examination of the grades recovered at +1.25 mm bottom screen size over the period indicate (Figure 14-6):

- Greater recovered grade variability over consecutive or isolated months for 100% M/PK(S) and 100% EM/PK(S) than can reasonably be explained by (unmitigated) country-rock dilution or by occasionally mistaken plant feed provenance, especially in the period prior to January 2020. These data support production-scale local intra-domain grade variations for both M/PK(S) and EM/PK(S);

- Remarkably consistent recovered grade *variation* for M/PK(S) in the period after January 2020, and centered on a mean recovered grade of 10.6 ± 2.0 cpht (± 1 standard deviation), some 2% below the 10.8 cpht global grade estimated for M/PK(S) in the 2023 MRE; and

- A tendency for EM/PK(S) to underperform the 21.0 cpht global grade estimated in the 2023 MRE. This is most obvious in the period after January 2020 during which M/PK(S) mean grade recovery remains consistent.

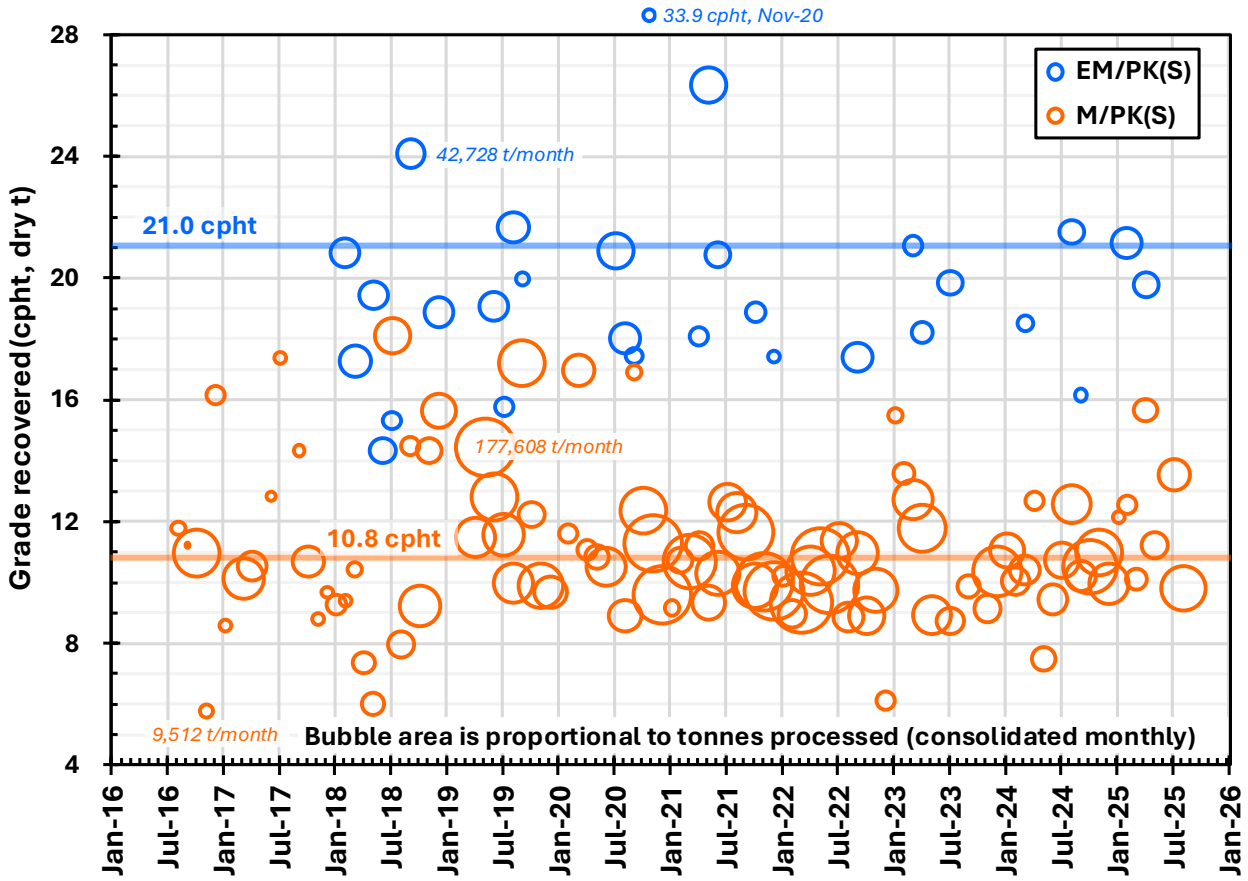
Table 14-3 provides a summary of the grade recovery data for the discrete production from M/PK(S) and EM/PK(S); the summary is split at January 2020 to isolate the consistent M/PK(S) mean grade recovery over 63 months since January 2020. The latter date also coincides with completion of material process plant modifications during 2018 and 2019.

Mean grades recovered for M/PK(S) and EM/PK(S) over the period 2020-2025 are in good agreement with global grades estimated in the June 2023 MRE, though with 1.9% under-recovery of mean global grades evident for both of M/PK(S) and EM/PK(S). The magnitude of under-recovery is compatible with minor diamond losses during milling and processing of kimberlite (e.g., Coward, 2020). Actual diamond losses to process plant tailings have not been tested, estimated or quantified for the KDM.

In summary: Mean grades recovered at +1.25 mm bottom screen size from processing of 4,700 kt M/PK(S) and 553 kt EM/PK(S) in the period January 2020 to September 2025 fully support global resource grades of 10.8 cpht and 21.0 cpht, as respectively estimated for M/PK(S) and EM/PK(S) in the June 2023 MRE (Doerksen et al. 2024).

The continuity to depth of these global grades was established by independent QPs Ms. Webb and Mr. Revering (see Doerksen et al. 2024) by integrating core logging, petrographic and microdiamond assay data for materially the same M/PK(S) and EM/PK(S) domains with LDDH-derived macrodiamond data at OP depths that, in this work, have been ratified as consistent with mean OP grades recovered at KDM production scale. Dr Grütter accordingly endorses that the domain-based global grade continuity established in June 2023 remains valid and is fit for use in the September 2025 updated MRE.

Figure 14-6: Grade Recovered from Pure M/PK(S) or EM/PK(S) Plant Feed



Note:
Monthly consolidated data for select daily production. Grades as recovered at +1.25 mm bottom cut-off.
Source: SRK (2025)

Table 14-3: M/PK(S) and EM/PK(S) Recovered Grades for Discrete Production, 2016 to 2025

Item	Discrete Production Parcels			
	M/PK(S)	M/PK(S)	EM/PK(S)	EM/PK(S)
	2016-2019	2020-2025	2018-2019	2020-2025
Discrete production period (months, consolidated)	33	63	11	18
Headfeed (dry tonnes)	1,578,462	4,666,999	412,089	552,933
Diamonds Recovered (+1DTC, carats)	191,759	496,805	79,065	113,630
Mean Recovered Grade (cpht)	12.1	10.6	19.2	20.6

Item	Discrete Production Parcels			
	M/PK(S)	M/PK(S)	EM/PK(S)	EM/PK(S)
	2016-2019	2020-2025	2018-2019	2020-2025
Grade Standard Deviation (cpht, monthly)	3.2	2.0	2.9	4.1
Grade Standard Error (cpht, monthly)	0.6	0.2	0.9	1.0
June 2023 MRE Global Grade (cpht)		10.8		21.0
% Global Grade not recovered		1.9%		1.9%

Note:
+1.25 mm bottom cut-off applies to Mean Recovered Grade and June 2023 Global Grade.
Source: SRK (2025)

14.4.3 2025 Updated Size Frequency Distributions

The size frequency distribution (SFD) of diamonds recovered from discrete production of South Lobe M/PK(S) and EM/PK(S) over the period January 2016 to September 2025 is provided in Table 14-4. Minor variations are evident in the proportion of diamonds reporting to a specific sieve class; the EM/PK(S) produces proportionately more diamonds larger than the DTC#21 sieve (i.e., >3.7 ct rough) than M/PK(S).

It is noted that modifications within the KDM process plant over the course of 2018 and 2019 were aimed at improving the recovery of diamonds smaller than the DTC#7 sieve, though no such general improvement in small diamond recovery is evident in the discrete production data. However, the recovery of DTC#5-7 class diamonds (0.05 to 0.12 ct rough) did improve after January 2020.

A complete comparison of SFDs in Table 14-4 with June 2023 SFDs is not possible due to differences in diamond size classes then used, but a partial comparison for the largest and smallest diamond size classes is presented in Table 14-5. The partial comparison illustrates the subtly variable SFDs that have been recorded through time for the M/PK(S) and EM/PK(S), and that model SFDs developed during estimation of the June 2023 MRE are generally concordant with those recorded from discrete production over the period January 2016 to September 2025.

Table 14-4: Diamond Size Frequency Distributions for Discrete Production, 2016 to 2025

	Discrete Production Parcels				Discrete Production Parcel SFD's			
	(ct per size class)				(% ct per size class)			
Size Class	M/PK(S)	M/PK(S)	EM/PK(S)	EM/PK(S)	M/PK(S)	M/PK(S)	EM/PK(S)	EM/PK(S)
(DTC)	2016-2019	2020-2025	2018-2019	2020-2025	2016-2019	2020-2025	2018-2019	2020-2025
+10.8 ct	12,132	34,993	5,046	11,610	6.3	7.0	6.4	10.2
+23	3,276	7,712	1,407	2,006	1.7	1.6	1.8	1.8
+21	8,865	19,922	3,761	4,891	4.6	4.0	4.8	4.3
+19	13,045	32,139	4,933	6,500	6.8	6.5	6.2	5.7
+17	8,088	20,840	2,838	4,124	4.2	4.2	3.6	3.6
+15	5,660	14,990	2,108	2,988	3.0	3.0	2.7	2.6
+13	18,356	48,119	7,309	9,676	9.6	9.7	9.2	8.5
+12	13,694	32,647	5,162	6,542	7.1	6.6	6.5	5.8
+11	21,254	57,319	9,239	12,159	11.1	11.5	11.7	10.7
+9	28,563	77,030	12,053	16,798	14.9	15.5	15.2	14.8
+7	20,255	53,392	8,537	12,187	10.6	10.7	10.8	10.7
+5	32,850	87,058	14,116	21,205	17.1	17.5	17.9	18.7
+3	5,591	10,448	2,499	2,897	2.9	2.1	3.2	2.5
+1	132	196	55	46	0.1	0	0.1	0
-1	0	0	0	0	0	0	0	0
Total Carats	**191,759**	**496,805**	**79,065**	**113,630**	**100**	**99.9**	**100.1**	**99.9**
% ct > 21 DTC					12.6	12.6	13.0	16.3
% ct < 7 DTC					20.1	19.6	21.2	21.2

Source: SRK (2025)

Table 14-5: Comparison of South Lobe SFD (partial)

	Model SFD's		Discrete Production Parcel SFD's			
	(% ct per size class)		(% ct per size class)			
Size Class	M/PK(S)	EM/PK(S)	M/PK(S)	M/PK(S)	EM/PK(S)	EM/PK(S)
(DTC)	2023 MRE	2023 MRE	2016-2019	2020-2025	2018-2019	2020-2025
+10.8 ct	6.3	8.0	6.3	7.0	6.4	10.2
+9	15.2	14.4	14.9	15.5	15.2	14.8
+7	11.2	10.9	10.6	10.7	10.8	10.7
+5	15.3	18.9	17.1	17.5	17.9	18.7
+3	2.0	4.1	2.9	2.1	3.2	2.5
Total	**50.0**	**56.3**	**51.8**	**52.8**	**53.5**	**56.9**

Source: SRK (2025)

14.4.4 Grade Estimate for KIMB3

As no new drilling or sampling information is available for KIMB3, the material information underpinning the June 2023 global grade estimate of 10.8 cpht (+ 1.25 mm) for the KIMB3 domain has not changed. Dr Grütter endorses that “microdiamonds from KIMB3 have a similar SFD as microdiamonds from the M/PK(S) domain”, that “the ratio of micro- to macrodiamonds obtained for M/PK(S) material can be assumed applicable to KIMB3”; and that since “no bulk sampling of the KIMB3 domain has been conducted to date, a significant amount of uncertainty is associated with the macrodiamond grade projection for the KIMB3 domain” (quoted from Section 14.3.5 of Doerksen et al. 2024).

14.4.5 Grade Estimates for Other Minor Units

Section 10 of this Report outlines drilling completed in 2025Q3 to improve basic knowledge of geotechnical and related parameters for KIMB3 and other minor kimberlite units occurring in the South Lobe, predominantly below 600 masl. Geotechnical and geological attributes for KIMB3 and six additional minor kimberlite units (KIMB1, KIMB4a, KIMB5, KIMB6, WM/PK(S), RFW-ALT) were constrained by logging of drill cores, and sampling of core for petrography, density and microdiamond assay (pre-2025 drillholes). A summary of the microdiamond data for these units is provided in Table 14-6, together with their diamond carrying capacity, relevant dilution characteristics and their petrographic similarity to the major kimberlite units in the South Lobe.

Diamond carrying capacity (DCC) refers to the potential of a kimberlite to have sampled and preserved deep mantle material and by inference has the potential to contain a commercially viable concentration of diamonds. It is an important qualitative and semi-quantitative assessment used during diamond exploration and resource evaluation where higher diamond carrying

capacities and higher grade potential are typically associated with high microdiamond counts, high olivine macrocryst content, high mantle (indicator) mineral content, low crustal dilution and medium- to coarse-grained rock textures (e.g., Nowicki et al., 2007).

Dr Grütter has reviewed all available microdiamond data and DCC considerations and estimates conservative recoverable macrodiamond grades for the six minor units as tabulated in Table 14-6. While this exercise reflects expert opinion, it nevertheless results in low-confidence, permissive grade estimates, and would be impossible to complete without the known benchmarking characteristics for the EM/PK(S), M/PK(S) and KIMB3 domains (Grütter, 2019).

Table 14-6: Microdiamond Content and Grade Estimates for Minor Kimberlite Units

Domain (Sep 2025)	Dry Mass (kg)	Total Stones (+105 µm)	Stones/kg (+150 µm)	Resource Grade (cpht +1.25 mm)	Considerations (Sept 2025)	Sept 2025 Estimated Grade (cpht +1.25 mm)
EM/PK(S)	4021.2	2468	0.39	21.0	Benchmark EM/PK(S) resource grade, microdiamonds per kg and diamond carrying capacity (DCC)	N/A
M/PK(S)	2649.9	920	0.21	10.8	Benchmark M/PK(S) resource grade, microdiamonds per kg and DCC	N/A
KIMB3	313.7	158	0.28	10.9	Benchmark KIMB3 resource grade, microdiamonds per kg and DCC	N/A
KIMB1	197.2	121	0.42		Higher DCC than KIMB3	15
KIMB4a	153.0	82	0.34		Similar DCC as EM/PK(S)	15
KIMB5	114.5	49	0.30		Similar DCC as M/PK(S)	10
WM/PK(S)	55.8	32	0.36		Similar DCC as M/PK(S); 50% diluted	5
RFW-ALT	7.9	6	0.38		Similar DCC as KIMB3	10
KIMB6	7.8	3	0.26		Similar DCC as EM/PK(S)	15

Source: SRK (2025)

14.5 Diamond Value Estimate

Diamond value estimates presented in this section have been generated by Lucara and are based primarily on LOM production and sales information to the end of December 2024. Partial sales records up to end-September 2025 have been considered at lower weight as they reflect an unsold status for larger diamonds due to in-progress stone evaluation and/or prevailing diamond market conditions. The diamond value estimates incorporate trends observed over the past two to four years based on proprietary in-house analysis of data from three sales channels utilised by Lucara: HB Antwerp, diamond tenders and the Clara platform.

Dr. Grütter reviewed the results of a significant recompilation and update of Karowe sales data that was completed during February 2025 and was reported in March 2025 by way of the Lucara 2024 Annual Information Form (Lucara 2025). A further review of the sales data up to September 2025 occurred in October 2025, coincident with development of a proprietary revenue model for >US$5 million legacy diamonds derived from KDM (SRK 2025). Dr Grütter considers the information and market analysis provided by Lucara to reflect industry standards and sales practices and considers them to be reliable and consistent with the average US$ per carat prices disclosed in recent Lucara quarterly and annual financials.

14.5.1 Average US$/carat Prices

Diamond value estimates are the product of the size frequency distribution of a given diamond population (e.g., Table 14-4) and the diamond quality characteristics of that population; and are typically unique for each kimberlite domain within a deposit. Diamond pricing for the KDM 2025 Mineral Resource Estimate relies on actual average diamond values obtained for discrete production from South Lobe M/PK(S) and EM/PK(S) kimberlite, based on tracking of discrete daily diamond production through consolidation into parcels for final sale (see Section 12.2.5).

--The evolution with time of average diamond prices (in nominal US$ per carat, US$/ct) for South Lobe kimberlite is summarised in Table 14-7. A material decrease in average US$/ct prices is shown for March 2025; this resulted from a significant recompilation and update of discretely tracked Karowe diamond production and sales data to December 31, 2024 that completed during February 2025 and was incorporated in the Lucara 2024 Annual Information Form (dated March 31, 2025). The same discretely tracked sales data to December 31, 2024 are now fully populated across all diamond categories and sales channels utilised by Lucara, and inform approximately 3% improved average prices of US$520/ct for M/PK(S) and US$695/ct for EM/PK(S) diamonds. Since partial sales records to end-September 2025 are not inconsistent with these average prices (on a like-for-like basis), the prices listed under September 2025 in Table 14-7 are deemed appropriate for use in the current Mineral Resource Estimate.

Table 14-7: Summary of Average US$/ct Prices for South Lobe Kimberlite

Domain	July 2019 MRE	June 2023 MRE	March 2024 2023 AIF	March 2025 2024 AIF	September 2025 MRE
M/PK(S)	631	707	707	507	520
EM/PK(S)	777	828	828	672	695
KIMB3	631	707	-	-	-

Note:
Nominal US$/ct, MRE is Mineral Resource Estimate, AIF is Annual Information Form.
Source: SRK (2025)

No price estimates are provided for kimberlite in the North and Centre Lobes since they have been relegated to an Unclassified status (see Section 14.1). Using 5% of monthly headfeed tonnes as a threshold, the most recent consequential production from the North Lobe occurred during February 2023, and during February 2025 for the Centre Lobe. Updated prices for stockpiled material from the North and Centre Lobes are reported in Section 15.

14.5.2 Revenue Model for >US$5 million Legacy Diamonds

The mining of kimberlite from the KDM has produced forty-nine (49) large, gem-quality diamonds that were recovered and sold in the period March 2013 to November 2024, for US$5 million or more each. These high-value gemstones are designated as Legacy diamonds by Lucara and the 49 recovered and sold over the 12-year period account for US$557.3 million in total revenue (in nominal US$). Lucara Diamonds has to-date not included this revenue in revenue models for the Karowe asset, on account of the infrequent and seemingly erratic recovery of these diamonds. Well-known (named) examples of Legacy diamonds from Karowe are listed in Table 14-8.

Table 14-8: A selection of named Legacy diamonds from Karowe

Diamond Name	Weight (ct, rough)	Remark
Constellation	813	Sold for US$63.1 million in May 2016 (Note 1)
Lesedi La Rona	1,109	Sold for US$53.0 million in September 2017 (Note 1)
Sethunya	549	Sold for US$35.0 million in November 2024 (Note 1)
Motswedi	2,488	Recovered August 2024; unsold (Note 2)
Seriti	1,094	Recovered September 2024; unsold (Note 2)

Notes:
1. The 2025 Legacy diamond revenue model is capped at US$25.62 million per stone.
2. The 2025 Legacy diamond revenue model excludes unsold diamonds.
Source: SRK (2025)

During September 2025, Dr Grütter was given confidential access to the Lucara Legacy diamond database. A selection of records were verified against public-domain disclosure, with no material discrepancies noted. A proprietary revenue frequency distribution (RFD) model for Legacy diamonds was developed and reported on in October 2025 (SRK 2025). Lucara has provided authorization to disclose the following aspects of the 2025 RFD model:

- A cap of US$25.62 million per Legacy stone was applied in the RFD model, resulting in 13.3% lower total (capped) revenue recognised than actual total sales of US$557.3 million (Table 14-9);
- The variance of the (capped) RFD model remains within ±0.5% to ±7% of (capped) annual revenues for 11 of the 12 years considered (2013-2024), implying that the RFD model suitably describes (capped) annual revenues generated by sporadic Legacy diamond recoveries; and
- When normalised to plant feed over the period 2015 to 2024, the RFD model attributes (capped) mean annual revenues of US$43.0 million per million tonne processed for the EM/PK(S) and US$13.7 million per million tonne processed for the M/PK(S).

Table 14-9: Summary of 2025 Revenue Frequency Distribution Model for Legacy Diamonds

Item	Period	M/PK(S)	EM/PK(S)	Centre Lobe	Total
Legacy Diamond Count (n =)	2013-2024	29	17	3	49
Actual Revenue (US$M)	2013-2024	255.4	284.9	17.0	557.3
Capped Revenue (Capped US$M)	2013-2024	255.4	210.6	17.0	483.0
RFD Model Revenue (Capped US$M)	2013-2024	261.9	207.5	-	
Revenue Variance (RFD Model / Capped Revenue, %)	2013-2024	+2.5%	-1.5%	-	
Mean Attributable RFD Model Revenue / Mt processed (Capped US$M / Mt)	2015-2024	13.7	43.0	-	

Note:

Mean Attributable RFD Model Revenue / Mt processed can be applied when reasonable prospects exist of processing more than 2.0 Mt of combined M/PK(S) and EM/PK(S). The potential exists of underperforming (or overperforming) Mean Attributable RFD Model Revenue at lower tonnage processed, to the extent that zero Legacy diamonds could be recovered.

Source: SRK (2025)

14.6 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 states that "reasonable prospects for eventual economic extraction" implies:

> *"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 2025 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 (760 m below surface), based on drillhole coverage, geological continuity and available sample information (i.e., petrographic 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 and six additional minor domains

(KIMB1, KIMB4a, KIMB5, KIMB6, WM/PK(S) and RFW-ALT) are entirely classified as Inferred Mineral Resources due to insufficient diamond data to support an assessment of macrodiamond grade and value characteristics within these kimberlite domains, and insufficient drillhole coverage to adequately assess geological continuity at higher confidence levels.

The North and Centre Lobes are relegated to an Unclassified status in the current work; they were both classified as Indicated Mineral Resources to a depth of 745 masl in the June 2023 MRE (Section 14.5 of Doerksen et al. 2024).

The 2025 Mineral Resource statement for KDM (with an effective date of September 30, 2025) is provided in Table 14-10 and is inclusive of Mineral Reserves.

Table 14-10: KDM 2025 Mineral Resource Statement (Effective September 30, 2025)

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mct)	Grade (cpht)	Average US$/ct
Indicated	South_M/PK(S)	5.28	15.70	2.97	1.70	10.8	520
	South_EM/PK(S)	5.97	17.44	2.92	3.66	21.0	695
Total Indicated		**11.25**	**33.13**	**2.95**	**5.36**	**16.2**	639
Inferred	South_M/PK(S)	0.1	0.30	2.97	0.03	10.8	520
	South_EM/PK(S)	1.39	4.16	3.00	0.87	21.0	695
	South_KIMB3	0.45	1.20	2.68	0.13	10.9	
	South_WM/PK(S)	0.13	0.33	2.62	0.02	5	
	South_KIMB4a	0.1	0.30	2.95	0.05	15	
	South_KIMB1	0.07	0.20	2.74	0.03	15	
	South_KIMB5	0.06	0.17	2.96	0.02	10	
	South_KIMB6	0.00	0.01	2.96	0.00	15	
	South_RFW-ALT	0.02	0.04	2.35	0.00	10	
Total Inferred		**2.32**	**6.72**	**2.9**	**1.15**	**17.2**	

Notes:

1. Prepared by Hermanus Grütter, P.Geo, Ph.D. of SRK Consulting (Canada) Inc.;
2. CIM definitions were followed for Mineral Resources;
3. Effective date of September 30, 2025;
4. All numbers have been rounded to reflect accuracy of the estimate;
5. Mineral Resources are stated above a +1.25 mm bottom cut-off;
6. Average diamond value estimates are based on diamond sales data provided by Lucara Diamond Corp.;
7. Mineral Resources are in situ and inclusive of in situ Mineral Reserves;
8. Mineral Resources are exclusive of all mine stockpile material;
9. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery;
10. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability; and
11. 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.

Source: SRK (2025)

Dr. Grütter 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 was developed to extract the economic portions of KDM Indicated Mineral Resources plus stockpiled ore. The mine plan extracts only the South Lobe of the three adjacent lobes of kimberlite (South, Centre, North). The South Lobe is planned to be mined through a combination of OP and UG mining methods. Stockpiled ore consists of a mix of all three kimberlite lobes (South, Centre, North).

The UGP plans to use a hybrid combination of conventional block caving with overhand shrinkage stoping. The mine will feature an extraction level with drawpoints, drawbells, and central tips for material handling at approximately 310 masl. Additional ore is mined below the extraction level down to 250 masl, as some ventilation and water management infrastructure is established within the South Lobe,

Mining methods are described in detail in Section 16.6.

All Mineral Reserves are classified as Probable.

15.2 Mining Dilution

15.2.1 OP Dilution

No OP dilution has been included in the OP reserve estimate as the remaining mining activities in the OP are primarily within kimberlite domains.

15.2.2 UG Dilution

A total UG dilution of 23%, or 5.3 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.

The end of the UG mine life is a result of high external dilution decreasing rock value below the economic cut-off values at the end of Q4 2037.

15.2.2.1 External Dilution

External dilution accounts for additional host rock material that is mined outside of the resource shape. This material has zero grade and value assigned to it. External dilution estimates have been defined by geomechanical caving simulations, as outlined in Section 16.3.8.

The large, continuous nature of the resource combined with generally good ground conditions in both the kimberlite and the host granite below 500 masl suggests minor dilution will occur where blast assisted caving operations are planned. External dilution in this region is primarily related to developing drawpoints in waste to increase the size of the draw footprint and improve overall mining recoveries.

At approximately 500 masl, or 170 m above the extraction level, unassisted block caving is planned. The majority of external dilution is associated with this region of unassisted block caving from both host rock in the vicinity to the resource as well as host rock material rilling from the open pit into the cave drawdown shape.

Geomechanical caving simulations have been used to estimate that 4.3 Mt of host rock is expected as dilution prior to mining operations stopping due to an economic cut-off at the end of the mine life.

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. All internal dilution is associated with Inferred resource material within the mining reserve cave drawdown shape. This material is treated as waste and has been assigned zero value.

Internal dilution of Inferred material comprises approximately 1 Mt of the UG reserve.

15.3 Mining Recovery

15.3.1 Open Pit Mining Recovery

A 95% mining recovery has been assumed for the OP reserves. This assumption has been applied to account for potential operational challenges in achieving final design limits. The mine design was recently updated to maximise extraction from the OP by steepening pit walls and mining deeper than previously planned.

15.3.2 Underground Mining Recovery

An UG mining recovery of 80% has been estimated at the end of the economic mine life in Q4 2037. UG mining recovery losses consist of two main components:

- Resource falling outside of the area of influence or cave drawdown shape; and
- Resource material inside the cave drawdown shape that is not extracted from the drawpoints prior to the end of the economic mine life. This is material that is either slow moving on the exterior of the shape, or heavily diluted material that is mobile but no longer economic.

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.

15.5 Cut-off Value Criteria

The kimberlite domains have varying diamond prices per carat as outlined in Section 14.5]. The diamond price per carat in the stockpile is determined from the weighted average of the North, Centre, South – M/PK(s) and South – EM/PK(S) domains. The value of legacy diamonds (>US$5 million) has been incorporated in the financial model and is in addition to the diamond price estimate. A COV by mining method is used to calculate the mining reserve, instead of determining a specific cut-off grade for each kimberlite domain.

Rock value is calculated from diamond valuation, payable content, royalties, mining dilution, mining recovery, and process recovery parameters.

Operating costs were estimated from existing operational budgets, previous studies, and future forecasts. The COV consists 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 all OP and UG mining operations.

The COV parameters are shown in Table 15-1.

Table 15-1: Cut-off Value Parameters

Parameter	Unit	Value		
		Open Pit	Underground	Stockpile[1]
Revenue, Smelting and Refining				
Payable content	%	100.0		
Royalty	%	10.0		
Mining Recovery and Dilution				
Mining Recovery	%	95	80	N/A
Mining Dilution	%	0	23	N/A
Processing Recovery	%	100		
Operating Costs				
Mining	US$/t milled	13.00	15.00	2.50

Parameter	Unit	Value		
		Open Pit	Underground	Stockpile[1]
Processing	US$/t milled	12.00		
G&A	US$/t milled	10.00	10.00	5.00
Cut-off Value	**US$/t milled**	**35.00**	**37.00**	**19.50**

Notes:

1. Stockpile costs apply when processed at the end of mine life. G&A costs are expected to remain at US$10.00/t when stockpile material is processed in the period between the completion of the open pit and commercial production of the underground.

15.6 Mineral Reserve Estimate

The Mineral Reserves identified in Table 15-2 complies with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, permitting, and other relevant factors are contained in the following sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

This study did not identify any mining, metallurgical, infrastructure, permitting, or other relevant modifying factors that may materially affect the estimates of the Mineral Reserves or potential production. The QP 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. All Mineral Reserves are classified as Probable. The effective date of this Mineral Reserve Estimate is September 30, 2025.

Table 15-2: Karowe Mine Mineral Reserve Estimate (September 30, 2025)

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	Price
		(Mt)	('000s ct)	(cpht)	(US$/ct)
Open Pit					
South - EM/PK(S)	Probable	0.1	25	21.0	695
South - M/PK(s)	Probable	1.0	109	10.8	520
Open Pit	**Total**	**1.1**	**134**	**11.9**	**553**
Underground					
South - EM/PK(S)	Probable	14.2	2,429	17.1	695
South - M/PK(s)	Probable	14.3	1,240	8.6	520
Underground	**Total**	**28.5**	**3,669**	**12.9**	**636**
Stockpile					
Mixed Stockpile	Probable	4.2	546	13.1	254
Life of Mine	Probable	3.5	198	5.7	442
Stockpile	**Total**	**7.6**	**745**	**9.8**	**304**

Lobe	Reserve Category	Ore Tonnage	Carats	Grade	Price
		(Mt)	('000s ct)	(cpht)	(US$/ct)
Combined					
All	**Total**	**37.3**	**4,547**	**12.2**	**579**

Notes:

1. Prepared by QP, 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;
4. The bottom elevation of the Probable Reserve is 250 masl;
5. Mineral Reserves are quoted above a +1.25 mm bottom cut-off diamond size and have been factored to account for diamond losses within the smaller sieve classes expected within the current configuration of the Karowe Mine Process Plant;
6. Average diamond price estimates are based on diamond sales data provided by Lucara Diamond Corp; prices were updated in 2025 for actual rough diamond sales over the period 2021 to end 2024, with partial sales data in 2025. The value of legacy diamonds (>US$5 million) has been incorporated in the financial model and are in addition to the diamond price estimate;
7. Tonnages are rounded to the nearest 100,000 tonnes, diamond grades are rounded to one decimal place to properly reflect the Reserve estimate accuracy. Rounding may cause minor discrepancies in the Mineral Reserve Estimate;
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 US$35/t based on an OP mining cost of US$13/t, a processing cost of US$12/t and a G&A cost of US$10/t;
10. Underground Mineral Reserves are estimated at a cut-off value of US$37/t based on an UG mining cost of US$15/t, a processing cost of US$12/t and a G&A cost of US$10/t;
11. OP dilution was assumed to be 0% as mining activities are primarily within the kimberlite. OP Mining Recovery of 95% has been assumed to account for potential challenges in achieving final bench elevations;
12. UG dilution assumptions were revised in 2025 as a result of geomechanical cave simulations. A total UG dilution of 23%, or 5.3 Mt has been included in the UG reserve estimate. Internal inferred domains are included as zero-grade waste. Cave drawdown simulations indicate that mine recovery will be 80% as of December 2037;
13. Stockpile Mineral Reserves are estimated at a cut-off value of US$19.50/t based on a rehandle cost of US$2.50/t, a processing cost of US$12/t and a G&A cost of US$5/t, when processed at the end of mine life;
14. Stockpile Reserves are not included in the Karowe 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. North lobe diamond price is estimated at US$155/ct; Centre lobe diamond price is estimated at US$210/ct.

15.6.1 Comparison to Previous Estimates

The UG Mineral Reserves have decreased in comparison to previous mineral reserve estimates. In order of significance, this variation is a result of the following updates in the 2025 Feasibility Study:

1. Geomechanical cave simulation estimates of UG mining recovery losses associated with resource material outside of the area of influence or cave drawdown shape;

2. Geomechanical cave simulation estimate of updated UG mining recovery losses associated with resource material inside the cave drawdown shape that is either slow moving on the exterior of the shape, or heavily diluted material that is mobile but no longer economic;

3. The updated 2025 block model has resulted in a minor reduction of resource carats, as well as a reclassification of some material from Indicated to Inferred domains; and

4. Updates in OP mining plans and ongoing open pit depletions, resulting in the transfer of underground reserves to the open pit.

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 providing an average annual 2.85 Mt of kimberlite feed to the mill, plus additional ore to surface stockpiles. The OP mine operation is expected to terminate in 2026, ending at an elevation of 665 masl. The mine currently has approximately 7.6 Mt of stockpiled reserves available for processing above the economic cut-off.

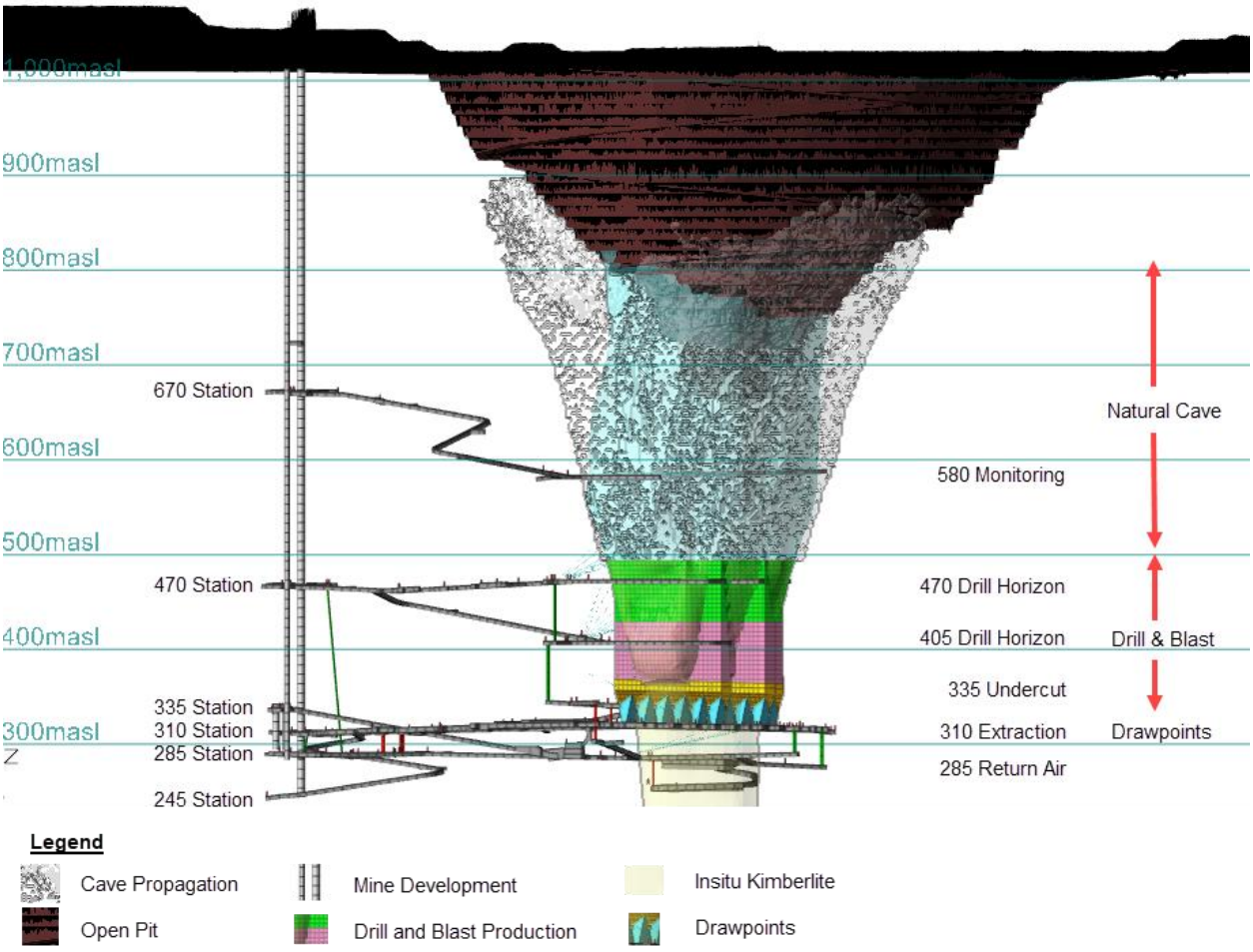
There are substantial resources remaining below the economic extents of the OP that may be extracted by UG mining methods. The mine plan proposes mineral extraction down to 250 masl, increasing mineral reserves by 28.5 Mt and mine life by 10 years.

The UGP plans to use a hybrid combination of conventional block caving with overhand shrinkage stoping. The mine will feature an extraction level with drawpoints, drawbells, and central tips for material handling. The first ~150 m above this level are mined bottom-up initially using an advanced undercut, followed by long-hole shrinkage methods, drawing only the swell. Higher up, as the orebody widens and becomes more favourable for caving, natural caving is induced. The mine operates in this manner until reserves are economically depleted. Figure 16-49 shows the mining sequence.

Construction of the UG Mine began in 2021 and is projected to reach a steady-state production target of 2.85 Mtpa in 2028, following a first principles, no-float development schedule. During the transition from OP to UG operations stockpiled OP ore will be processed, with 2.4 Mt of stockpile remaining for processing during mine closure in 2038.

The mine design and planning for KDM is based on the resource model completed by the QP, SRK in 2025, as detailed in Section 14 of this Report.

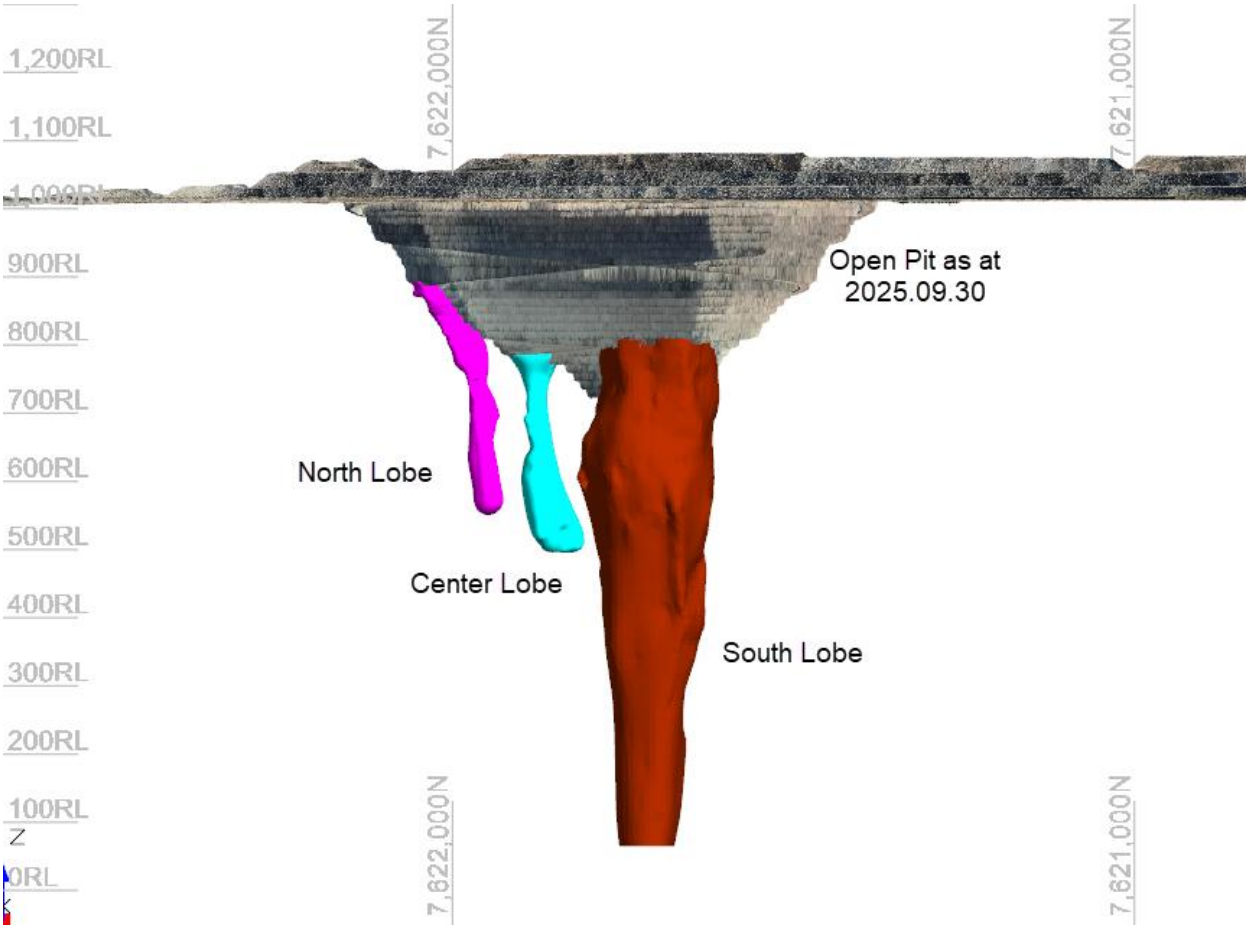
Figure 16-1: Underground Mine General Arrangement (Looking North)



16.2 Deposit Characteristics

KDM deposit contains three distinct coalescing pipes, referred to as the North, Centre, and South Lobes as illustrated in Figure 16-2. 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 not classified as a resource.

Figure 16-2: North, Centre, and South Kimberlite Lobe (Looking East)



Notes:
Elevations are stated in masl.

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

Table 16-1: South Lobe Dimensions and Hydraulic Radius

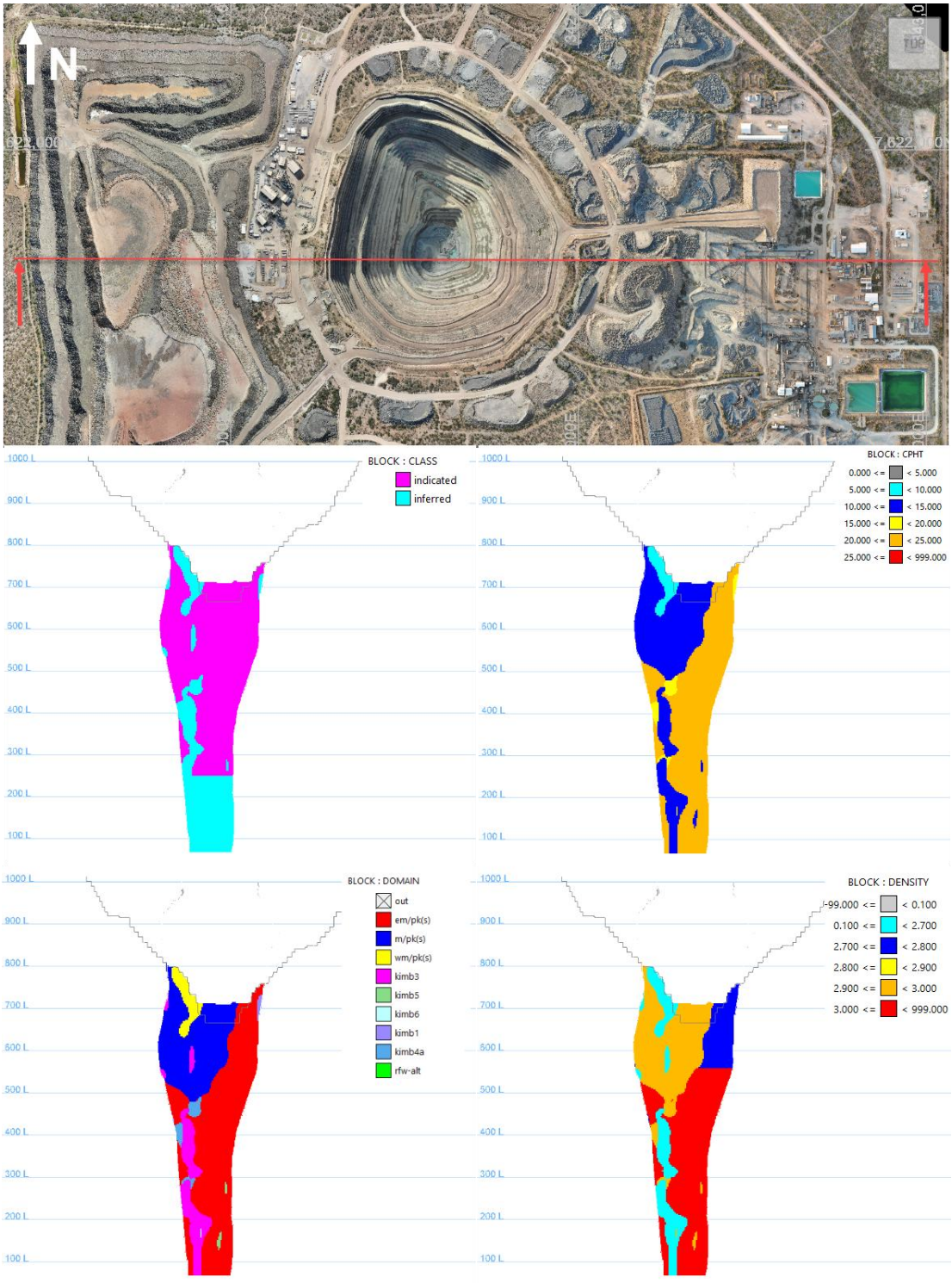
Elevation (mL)	Diameter (m)	Area (m^2)	Circumference (m)	Hydraulic Radius
800	215	35,593	702	51
700	207	33,236	659	50
600	213	35,113	703	50

Elevation (mL)	Diameter (m)	Area (m^2)	Circumference (m)	Hydraulic Radius
500	180	25,358	592	43
400	152	17,688	490	36
300	122	11,562	387	30
200	110	9,554	354	27
100	101	8,071	325	25

The South Lobe contains nine distinct domains, each with unique mineral properties. These domains are discussed in greater detail in Section 14. Two domains, namely EM/PK(S) and M/PK(S), make up more than 90% of the South Lobe resource and 100% of the Indicated reserves. The other eight domains are at an Inferred classification and, for reporting and economic modelling purposes, are 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 16-3 illustrates the South Lobe resources by domain, grade, classification, and density. By comparing the four figures, it becomes apparent that deeper resources contain higher grade at a greater tonnage factor, yielding more value per cubic metre of material mined.

Figure 16-3: South Lobe Resource Cross Section Looking North



16.3 Geotechnical Analysis

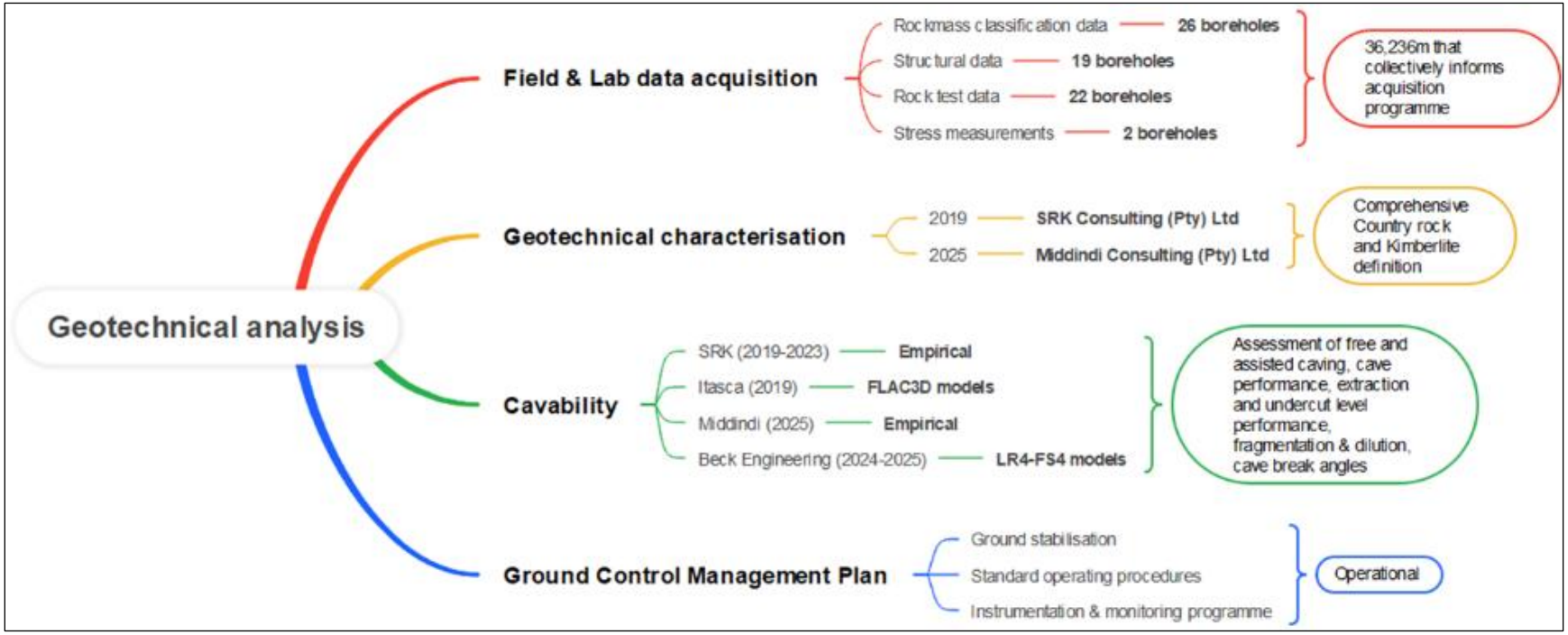
16.3.1 Introduction

The geotechnical component of the Karowe Diamond Mine design was completed through a collaborative program (2018-2025) of field investigations, laboratory testing, numerical modelling, and independent technical reviews (Figure 16-4). SRK Consulting Pty Ltd (South Africa) (SRK) oversaw the collection of geotechnical data, updated the initial geotechnical model, and provided shaft support specifications. Laboratory testing was carried out by RockLab (Pretoria, South Africa), while preliminary rock mass strength assessments and geomechanical analyses were undertaken by Itasca Consulting Group (Minneapolis, USA).

In 2025, the QP Middindi Consulting Pty Ltd (Middindi) was engaged to review and consolidate the geotechnical database into a single validated dataset (Middindi, 2025), and to evaluate the advanced numerical modelling conducted by Beck Engineering Pty Ltd (Beck) in 2025 (Beck, 2025). The empirical results obtained by the QP, together with Beck's large strain coupled discontinuum and hydro-mechanical models (LR4-FS4), confirmed the free and assisted cavability of the South Lobe kimberlite. The Ground Control Management Plan (GCMP) was revised in 2025 by the QP (Middindi, 2025) to reflect updated design methods, layouts, geotechnical interpretations and develop ground stabilisation in accordance with the latest revisions.

This geotechnical section of the Technical Report presents the outcomes of six years of consolidated and updated investigations. It describes inter alia an overview of the geotechnical database and testwork, assessments of rock mass quality, structural domains, weathering, stress conditions, and hydrogeological influences. Stability considerations are cited including cavability, cave performance and propagation, infrastructure stabilisation, subsidence risks, and dilution potential. The chapter concludes with a summary of risks and recommendations, providing a definitive foundation for mine design and the safe implementation of the UGP.

Figure 16-4: Overview of Concluded Geotechnical Studies



Source: Middindi (2025)

Chapter 28 lists all information sources used to compile this section. While the most recent reports (2025) form the primary basis for this Report, earlier reports, presentations, and analyses where relevant are also incorporated. The Technical Report draws on a wealth of geotechnical information developed and interpreted over the past six years.

16.3.2 Country Rock and Kimberlite Quality Characterisation

A number of drilling campaigns were concluded to obtain geological and geotechnical data to inform the current mine design. This includes the 2017-2018 drilling campaign, the 2019 geotechnical investigation program and the recent 2025 infill campaign. All drilling programs provided characterisation for both the country rock and South Lobe kimberlite. Geotechnical interpretations take cognisance of the updated 2025 geological model and geotechnical data up to the 2019 program. The number of boreholes, meterage, field and lab tests that supports the country rock and kimberlite geotechnical characterisation is listed in Table 16-2.

For details on GeoRock Codes and lithological abbreviations, refer to the Geology section of this Report.

Table 16-2: Geotechnical Data Acquisition Summary

Geotechnical Data	Country Rock	Kimberlite	Study Campaign
Number of boreholes	52	29 of 52	2017-2019
Meterage (m)	36,236.09	6,668.89	2017-2019
Validatory logging (BHs)	9	9	2025
Validatory meterage (m)	4,636.19	1,490.10	2025
Contact alteration (BHs)	25 (48 pierce points)		2025
Micro structural data (BHs)	19	19	2019
Structural pole count	19,514	7,182	2025
Rock tests	2,855	331	2019
Field PLT tests	6,662	1,216	2019

Source: Middindi (2025)

16.3.2.1 Country Rock (CR) Geotechnical Characteristics

The Kimberlite pipe between 1,014 masl and 665 masl is mined from an open pit. Primarily, discrete and intercalated sequences of sandstones and mudstones about the Kimberlite contact between 665 masl and 520 masl. From 520 masl to the extraction level (310 L) and below, the pipe is contained with granite basement. Selected rock quality (RMRL90) box and whisker plots for individual CR formations/lithologies are illustrated in Figure 16-5 (a full set are available in Middindi, 2026), and statistical results for RMRL90 and Q ratings listed in Table 16-3 and Table 16-4, respectively.

In accordance with the Laubscher RMRL90 system, the CR is predominantly “fair” rock mass with as expected occurrences of “poor” rock mass within the RED_MS1/2 (Red Mudstones), TLA_CMS2 (Carbonaceous Mudstones) and TLA_ARK1 (Porous Sandstones). An elevation distribution of the actual logged CR rock quality indicators is depicted in Figure 16-6. Fracture frequencies (FF/m) which are on average 3.5±2.66 within the CR is depicted in Figure 16-7. The following CR conclusions are drawn from the rock mass classification data:

The CR has on average a RMRL90 of 44±6 which classifies the rock mass as “fair” quality. The downgrade of quality to “poor” occurs within the red mudstones (36±4), carbonaceous mudstones (36±4) and porous sandstones (39±4). Level development (locally referred to as laterals) for blast assisted caving occurs exclusively in the granite basement. The 580 lvl contingency level occurs in the carbonaceous mudstones with sole purpose of providing an instrumentation and monitoring level and intermediate flood control level. Ground stabilisation for the 580 lvl has considered the “poor” rock formation;

Q ratings are collectively on average 18±12. The high standard deviation indicates large value spread (notwithstanding the log scale) and necessitates the use of the more conservative 25th percentile values for design purposes. All CR formations classify as “good” rock (mean = 12.34) with the exception of the mudstone and siltstone units (MOS_INT2, MOS_INT1, TLHAB_MS, TLA_CMS4) which have on average Q ratings of 7.2 (Fair rock);

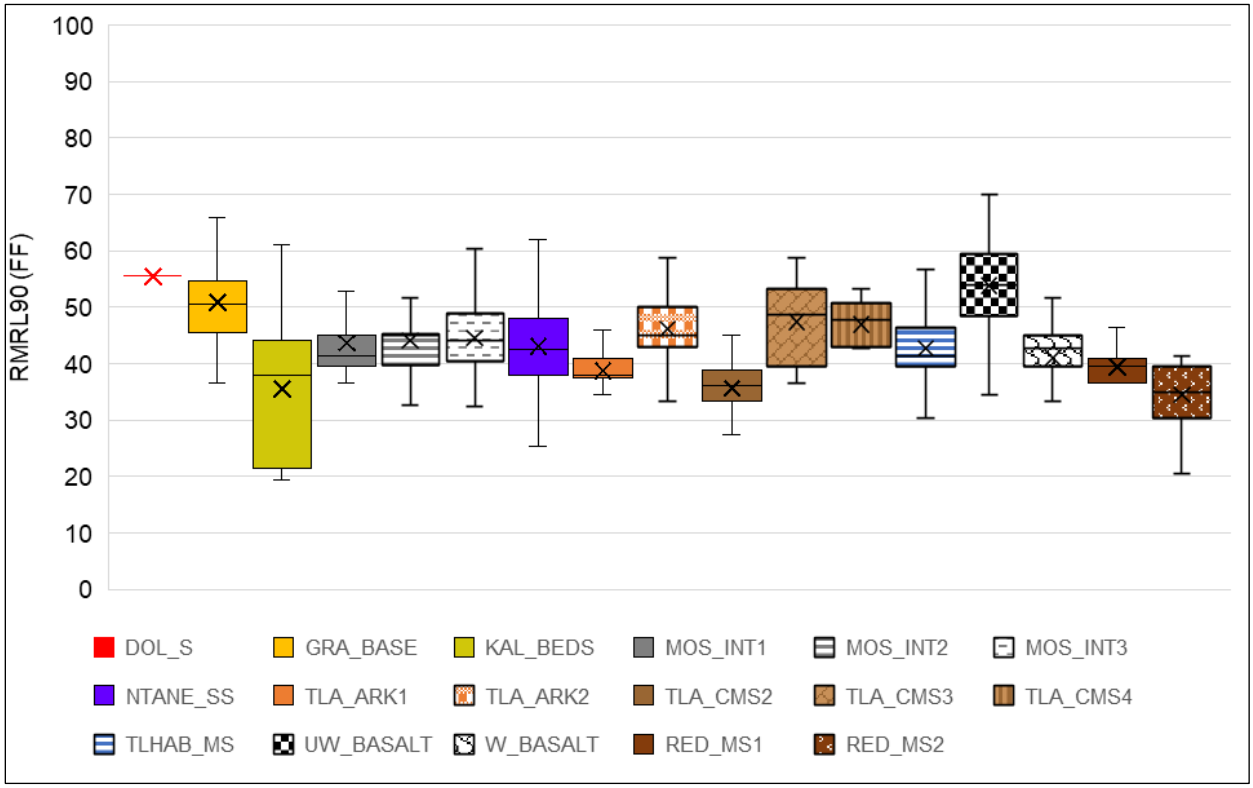
The granite basement indicates a marked departure point for improved rock quality indicators (Figure 16-8 and Figure 16-9) when compared to the overlying sediments. In excess of 50% of the South pipe will be extracted within basement material. All major LOM excavations have been sited within the granite basement (crusher chambers, loading pockets, flood control levels, access to extraction and undercut levels);

“Poor” rock mass conditions (RMRL90 < 40, Q < 4) are expected in the ±15 m of kaolinitised granite that caps the basement granite. No CR or Kimberlite development is planned within this horizon, but it remains a preferential conduit for water inflow into the pipe;

Open fracture frequencies for CR relevant to underground mining is typically low 3.55±2.08. Both closed and open fracture frequencies follow a similar trend from the pit to the base of the underground project, i.e., diminishing FF as depth increases. The granite basement has low FF of 2.56±1.61 indicative of superior rock mass conditions relative to the overlying sediments; and

Leached granite regions were identified within close proximity to the pipe, however, their location appears erratic across the UGP with the majority of the zones ranging from 0.04 m to 1.80 m in vertical extent. These zones (interconnectivity unknown) are suspected to exhibit significantly increased porosity and permeability compared to fresh granite, making them potential conduits for groundwater flow and storage, elevated pore pressures, and increased water ingress into the cave. The impact of leached granite zones on excavation stability will be minimal but 11 intercepts ±10 m from the 405 lvl drilling horizon suggests that water discharge into this level may need to be monitored and controlled (curtain grouting and dewatering).

Figure 16-5: RMR_{L90} (FF) for the Country Rock (CR)



Source: Middindi (2025)

Table 16-3: RMRL90 Statistics for the Country Rock

Formation	RMRL90 (FF)						
	min	mean	max	Std Dev	75th Percentile	25th Percentile	50th Percentile
KAL_BEDS	19	36	61	12	44	21	38
W_BASALT	20	41	59	8	45	39	43
UW_BASALT	31	54	70	8	60	48	54
NTANE_SS	14	43	69	7	48	38	42
MOS_INT3	21	45	68	7	49	40	44
MOS_INT2	33	44	71	7	45	40	45
MOS_INT1	36	44	57	6	45	39	41
TLHAB_MS	25	43	61	6	46	39	41
TLA_CMS4	43	47	53	4	51	43	48
TLA_CMS2	22	36	45	4	39	33	36

Formation	RMRL90 (FF)						
	min	mean	max	Std Dev	75th Percentile	25th Percentile	50th Percentile
GRA_BASE	36	51	77	7	55	45	50
TLA_ARK2	29	46	66	7	50	43	45
TLA_ARK1	28	39	48	4	41	37	38
TLA_CMS3	37	47	59	8	53	40	49
DOL_S	55	55	55	0	55	55	55
RED_MS1	36	39	46	3	40	36	39
RED_MS2	18	32	39	6	37	28	33

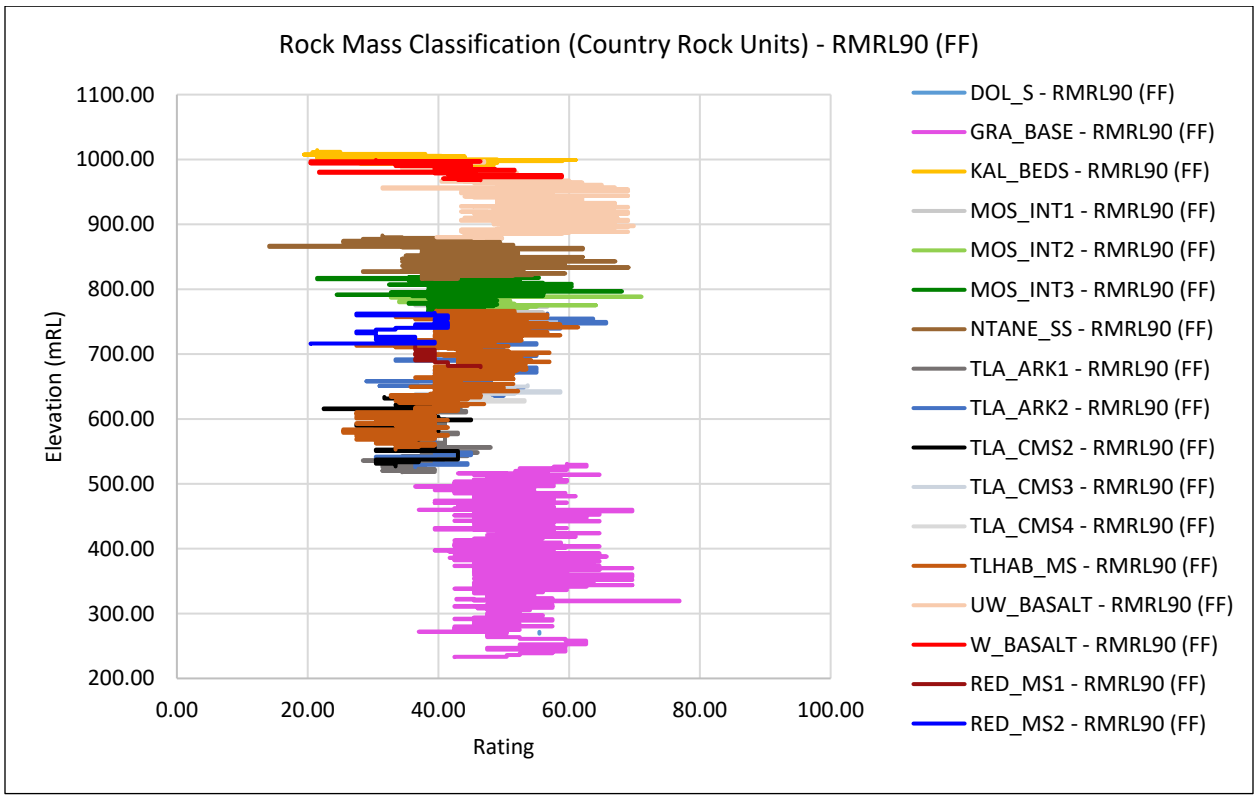
Source: Middindi (2025)

Table 16-4: Q-index Statistics for the Country Rock

Formation	Q-Index						
	min	mean	max	Std Dev	75th Percentile	25th Percentile	50th Percentile
KAL_BEDS	0.06	4.75	73.00	17.71	24.73	1.12	5.51
W_BASALT	0.39	13.34	48.83	13.00	27.88	5.67	16.79
UW_BASALT	2.45	34.67	150.00	46.28	71.38	18.75	25.00
NTANE_SS	0.44	14.70	56.32	10.23	21.94	10.25	16.50
MOS_INT3	1.47	16.72	132.00	14.99	25.00	11.09	16.50
MOS_INT2	3.50	13.32	43.34	10.13	24.85	8.32	13.04
MOS_INT1	4.88	10.18	40.18	7.05	12.87	7.69	8.90
TLHAB_MS	1.17	13.12	57.05	11.22	22.22	8.76	12.90
TLA_CMS4	4.07	9.38	21.56	8.52	20.96	4.10	9.82
TLA_CMS2	0.28	10.63	83.00	11.74	17.93	10.15	14.23
GRA_BASE	1.95	25.40	149.68	16.90	37.00	17.25	25.00
TLA_ARK2	4.41	25.24	63.65	11.26	32.73	23.11	32.26
TLA_ARK1	4.88	19.56	60.70	13.28	31.53	11.56	28.98
TLA_CMS3	11.50	24.26	48.67	13.84	36.61	13.76	29.82
DOL_S	16.56	16.56	16.56	0.00	16.56	16.56	16.56
RED_MS1	11.92	20.86	25.00	4.99	25.00	20.41	24.00
RED_MS2	3.62	13.99	25.00	7.70	23.93	10.13	15.50

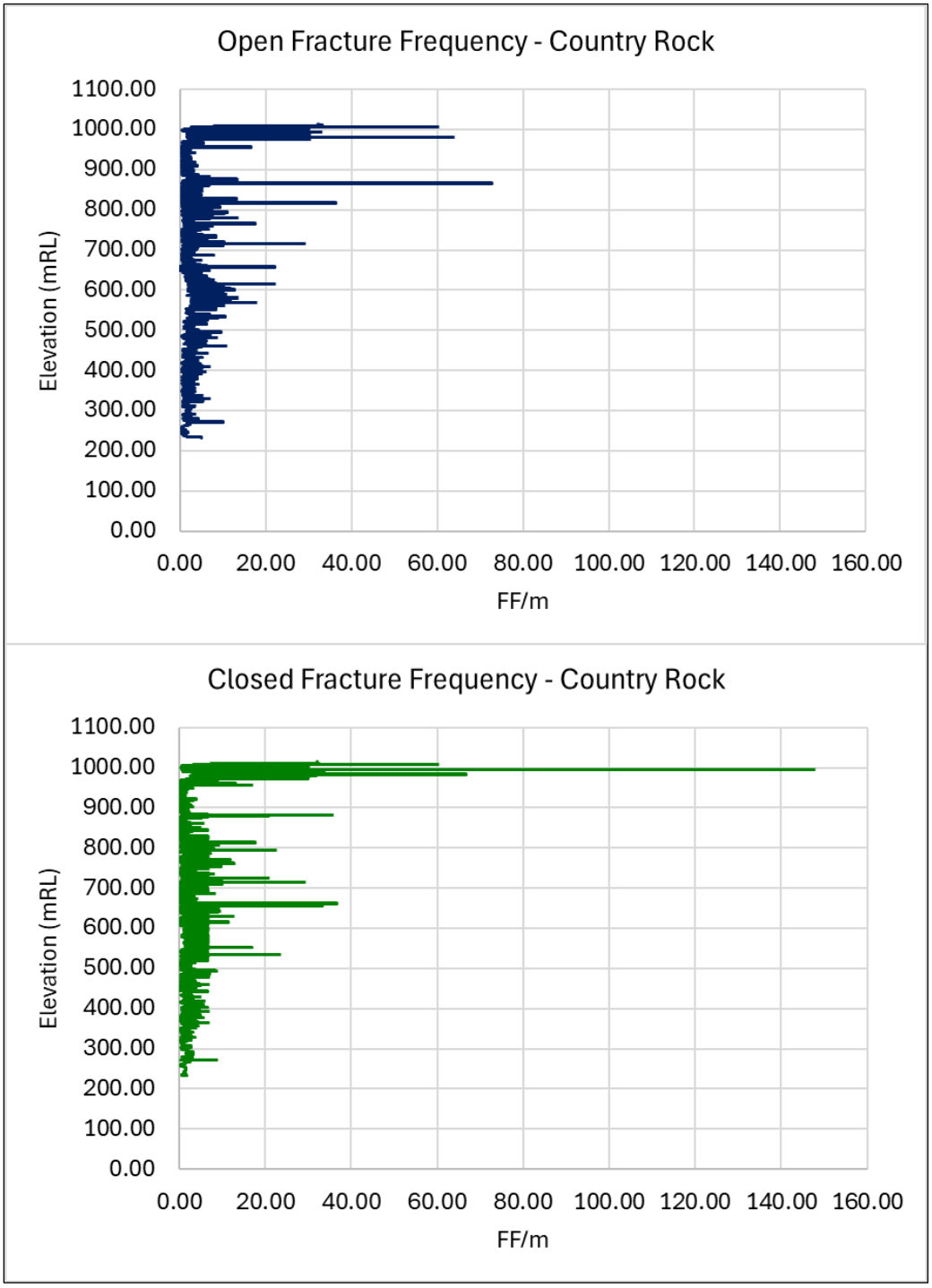
Source: Middindi (2025)

Figure 16-6: Rock Mass Classification Results – Country Rock with Elevation



Source: Middindi (2025)

Figure 16-7: Open and Closed Fracture Frequency – Country Rock



Source: Middindi (2025)

16.3.2.2 Kimberlite (KIMB) Geotechnical Characteristics

Rock quality indicators (RMR, MRMR, GSI & RMRL90 and Fracture frequency).

Selected rock quality (RMRL90) box and whisker plots for the South Lobe predominant KIMB facies are illustrated in Figure 16-8 and statistical results for RMRL90, and Q ratings listed in Table 16-5 and Table 16-6, respectively.

In accordance with the Laubscher RMRL90 system, the KIMB2/4 units which constitute >80% of the KIMB pipe is classified as “good” and KIMB3 as “fair” rock mass. An elevation distribution of the actual logged KIMB rock quality indicators is depicted in Figure 16-9. Fracture frequencies (FF/m) which are on average 2.54 ±2.81 are depicted in Figure 16-10.

The following conclusions are drawn from the rock mass classification data:

- The KIMB has on average a RMRL90 of 58.21±8 which classifies the rock mass as “fair” quality. The predominant KIMB2 (M/PK(S)) and KIMB4 (EM/PK(S)) both classify as “good” rock mass with ratings of 64.33 ±8.9;

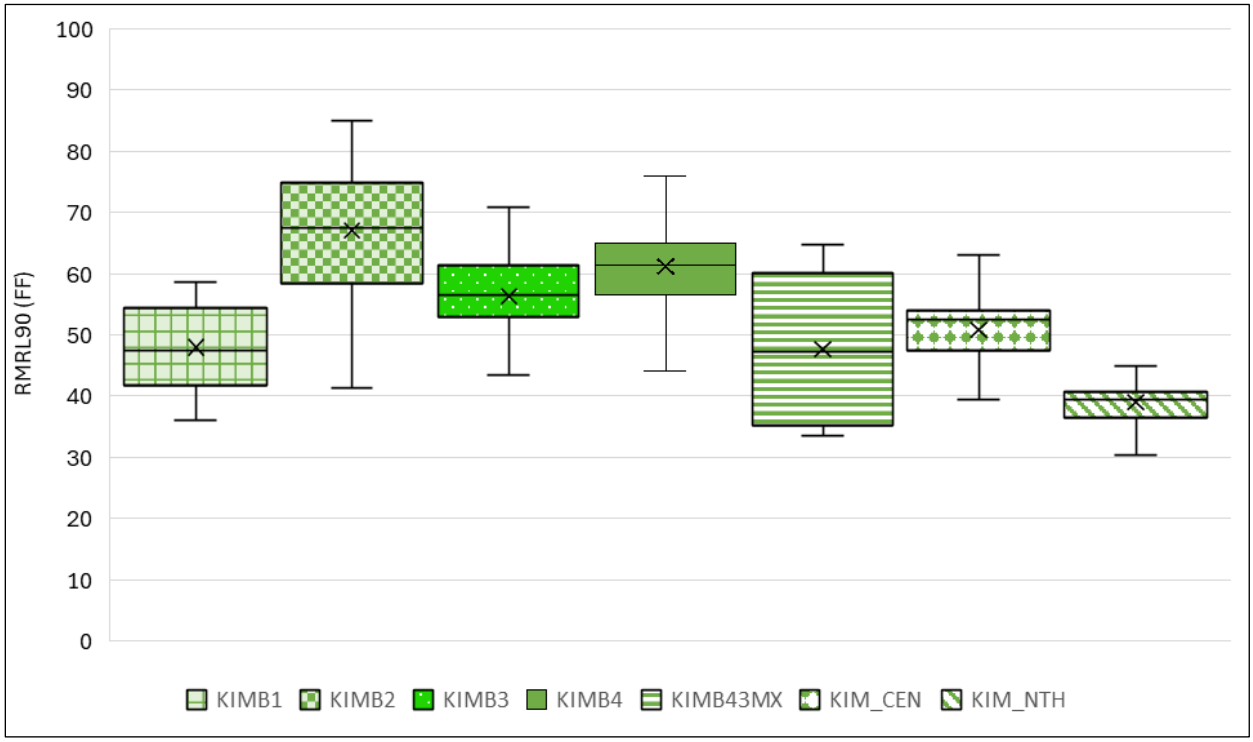
- Mean Q ratings are collectively on average 28.17 ±12 with 25th percentiles maintaining high values of 19.50. Both the RMRL90 and Q ratings provide “good” classifications; and

- Four distinct zones of high fracture frequency, which are analogous to disturbed rock mass resulting from secondary alteration, have been identified within the KIMB pipe (Figure 16-10). At least 22 m of vertical extent of high FF (>10/m) occurs on and between the extraction level (310 lvl) and undercut level (335 lvl). A second distinct band of high FF occurs 20 m immediately below the 405 lvl drilling horizon. The remaining two high FF bands are insignificant to underground mining since they occur outside the extractable battery limits.

During the design process discussions centred on mitigating the risk of high FF zones and localised alteration of a small area of the KIMB3 in the northwest area of the footprint, particularly between the extraction and undercut levels. The options raised were to (1) relocate the extraction zone above the high FF zone (approximately 345 masl) or (2) upgrade ground support to cater for expected “poor” ground conditions. The latter decision has been selected since (1) only seven boreholes intersected the extraction horizon (and some have incomplete geotechnical records) and hence the level of confidence in the extent of “poor” ground conditions was ill-defined; (2) fit for purpose ground support would be sufficient to mitigate extraction level conditions, in particular draw bell and associated pillar stability; (3) the relocation of the extraction level will erode project value.

Nevertheless, the stability of the extraction level remains a project risk. A 2025 drilling program aimed at creating better infill geotechnical knowledge of deeper sections of the pipe was postponed to 2026. Also, pillar and drawbell stability, although part of the 2024-2025 numerical modelling assessment, was not the focus of the study which concentrated on a cave performance assessment.

Figure 16-8: Rock Quality Indicators for the South Lobe Kimberlite (KIMB)



Note:
KIMB2 = M/PK(S); KIMB4 = EM/PK(S)

Source: Middindi (2025)

Table 16-5: RMRL90 Statistics for KIMB

Formation	RMRL90 (FF)						
	min	mean	max	Std Dev	75th Percentile	25th Percentile	50th Percentile
KIMB1	36	48	59	8	54	42	47
KIMB2	41	67	85	10	75	58	67
KIMB3	34	56	71	7	61	53	56
KIMB4	40	62	83	8	65	56	61
KIM_NTH	30	39	45	3	41	36	39
KIM_CEN	31	51	63	6	54	47	52

Note:
KIMB2 = M/PK(S); KIMB4 = EM/PK(S)

Source: Middindi (2025)

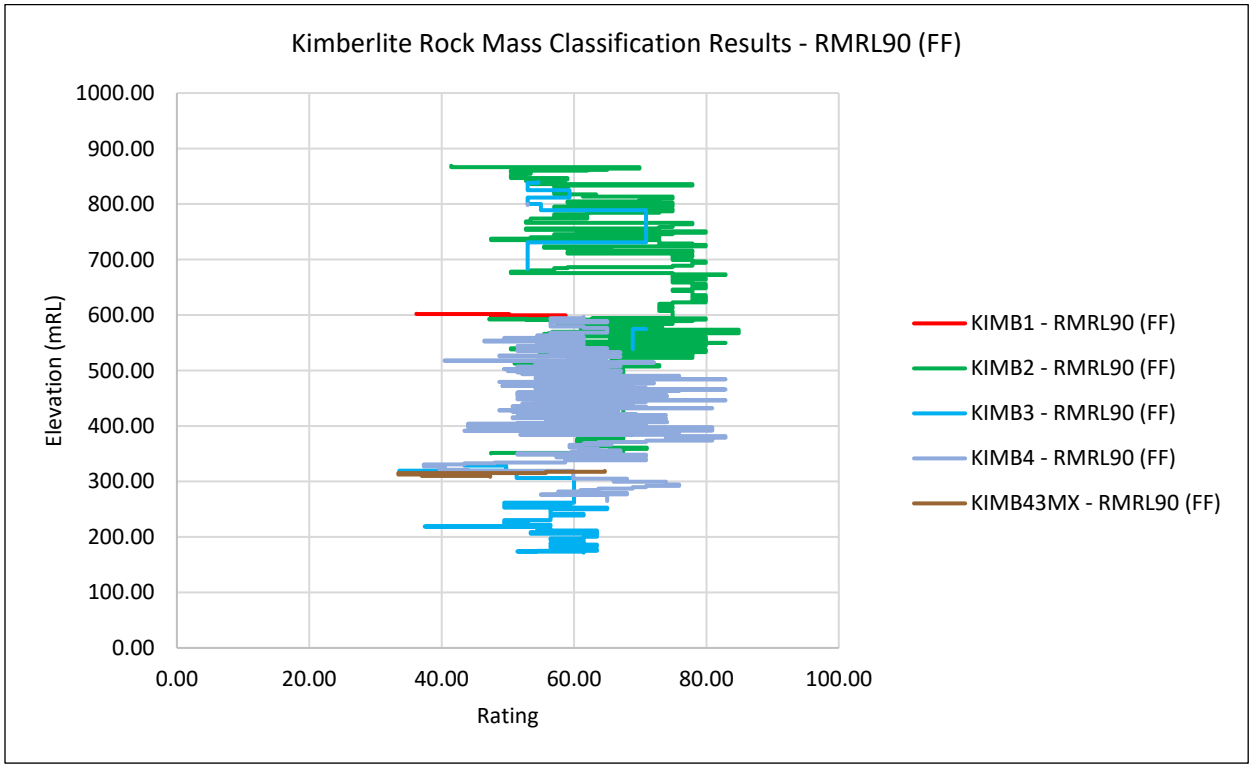
Table 16-6: Q-index Statistics for KIMB

Formation	Q-Index						
	min	mean	max	Std Dev	75th Percentile	25th Percentile	50th Percentile
KIMB1	0.42	10.60	70.43	27.09	47.72	5.76	16.42
KIMB2	1.38	35.29	99.63	34.18	91.44	18.61	48.15
KIMB3	0.39	31.10	99.45	21.25	37.50	30.38	36.05
KIMB4	1.13	35.70	100.00	27.34	60.67	23.25	46.14
KIM_NTH	3.92	25.62	45.24	10.76	35.54	27.57	30.50
KIM_CEN	2.88	24.26	47.50	12.40	36.03	23.70	31.44

Note:
KIMB2 = M/PK(S); KIMB4 = EM/PK(S)

Source: Middindi (2025)

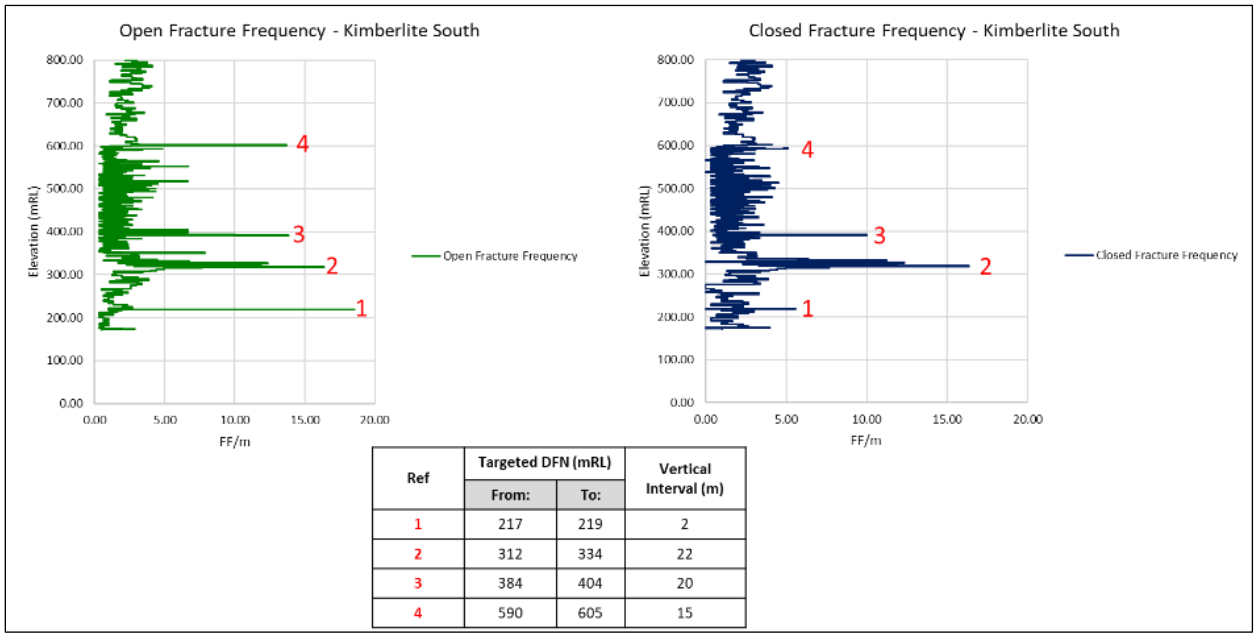
Figure 16-9: Rock Mass Classification Results – Kimberlite Facies



Note:
KIMB2 = M/PK(S); KIMB4 = EM/PK(S)

Source: Middindi (2025)

Figure 16-10: Open and Closed Fracture Frequency – KIMB with Elevation (n=9 BHs, 1490 m logged)



Ref	Targeted DFN (mRL)		Vertical Interval (m)
	From:	To:	
1	217	219	2
2	312	334	22
3	384	404	20
4	590	605	15

Source: Middindi (2025)

16.3.3 Rock Properties

16.3.3.1 Testing Database

An extensive laboratory and field rock testing program was completed between 2017-2019 (Table 16-7) which collectively included 331 KIMB and 2855 CR lab tests and collectively 7878 field index tests (point loads). The laboratory tests included Uniaxial Compressive Strength with elastic modulus (UCS/M), Triaxial Compressive Strength with and without elastic moduli (TCS/TCM), Unconfined Tensile Strength (UTB), and Shear Strength tests of Joints (STJO). Point load tests (PLT) were performed both axially and diametrically. The large number of test results allowed well defined representation of rock properties for both the CR and KIMB. All field and laboratory tests, logging and orientation data resides in a well organised database.

Table 16-7: Number and Distribution of Rock Tests (CR+KR)

Parameter	UCM	TCM	TCS	UTB	STJO	PLT	Total
Total Intact Tests (CR + KR)	707	94	1436	865	84	7878	11064
Country Rock (CR)	610	84	1305	782	74	6662	9517

Parameter	UCM	TCM	TCS	UTB	STJO	PLT	Total
Kimberlite South Lobe Intact	97	10	131	83	10	1216	1547
Percentage of Kimberlite (%)	13.72	10.64	9.12	9.60	11.90	15.44	13.98

Source: Middindi (2025)

16.3.3.2 Rock Strengths

In accordance with the ISRM[1] (1981) rock strength classification scale, the following points relate to CR and KIMB rock strengths (Table 16-8):

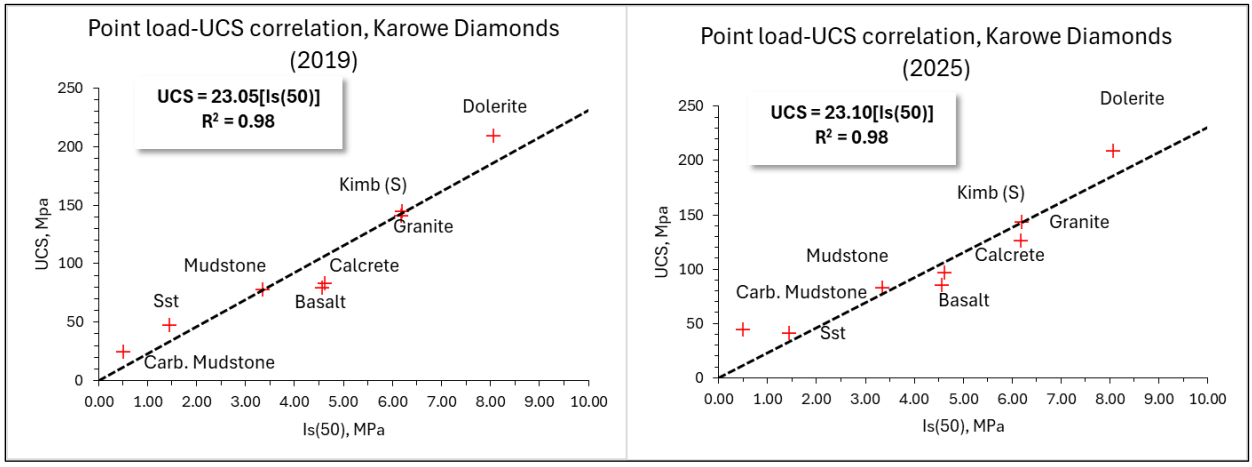
- The lowest mean UCS's (25-50 MPa) are recorded for predominantly the mudstone and siltstone units (NTANE_SS, MOT_INT1/2, RED_MS1/2 and TLA_ARK1 and TLA_CMS1);
- Remaining CR formations and lithologies class as "strong" (50-100 MPa);
- The KIMB1-4 facies are distinctly stronger than the CR with average UCS's of 138.71±36 MPa. The ISRM classification is universally "Very strong" rock; and
- Very good PLT and UCS correlations exist (Figure 16-11) providing the following conversion equation:

$$UCS_{MPa} = 23.10(Is_{50})$$

Similarly, the correlation between the UCS/TCS of basalt/kimberlite and density is exceptionally good but poor for the CR and granites (Figure 16-12). The KIMB is suited to the use of a density-UCS model to provide mine-scale strength definition; this approach was adopted by Beck (Beck, 2025) in their cave performance numerical simulations. Pipe vertical sections comparing the UCS distribution using UCS results alone against UCS converted from density is illustrated in Figure 16-13.

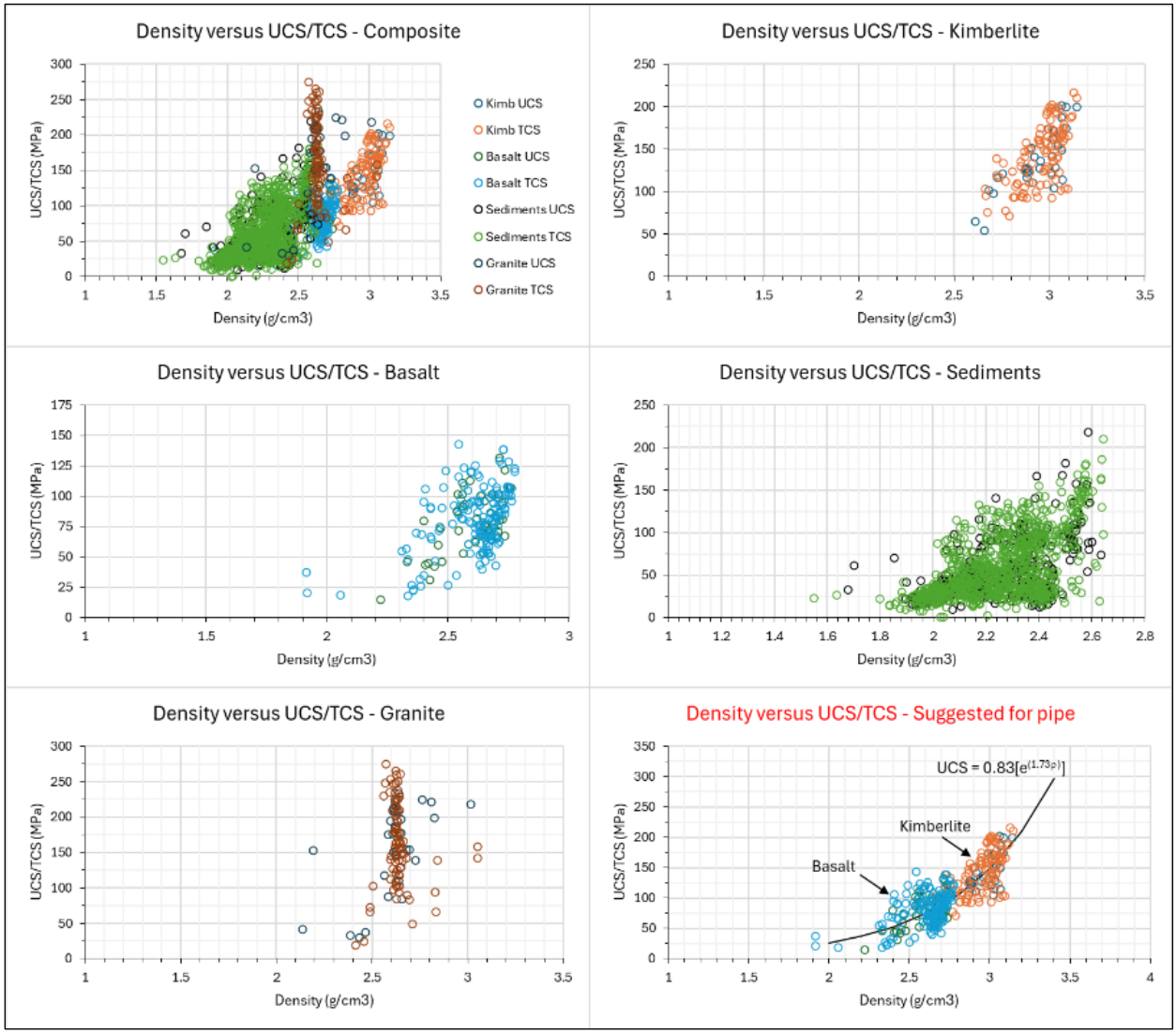
[1] International Society for Rock Mechanics

Figure 16-11: Point Load and Strength Correlations



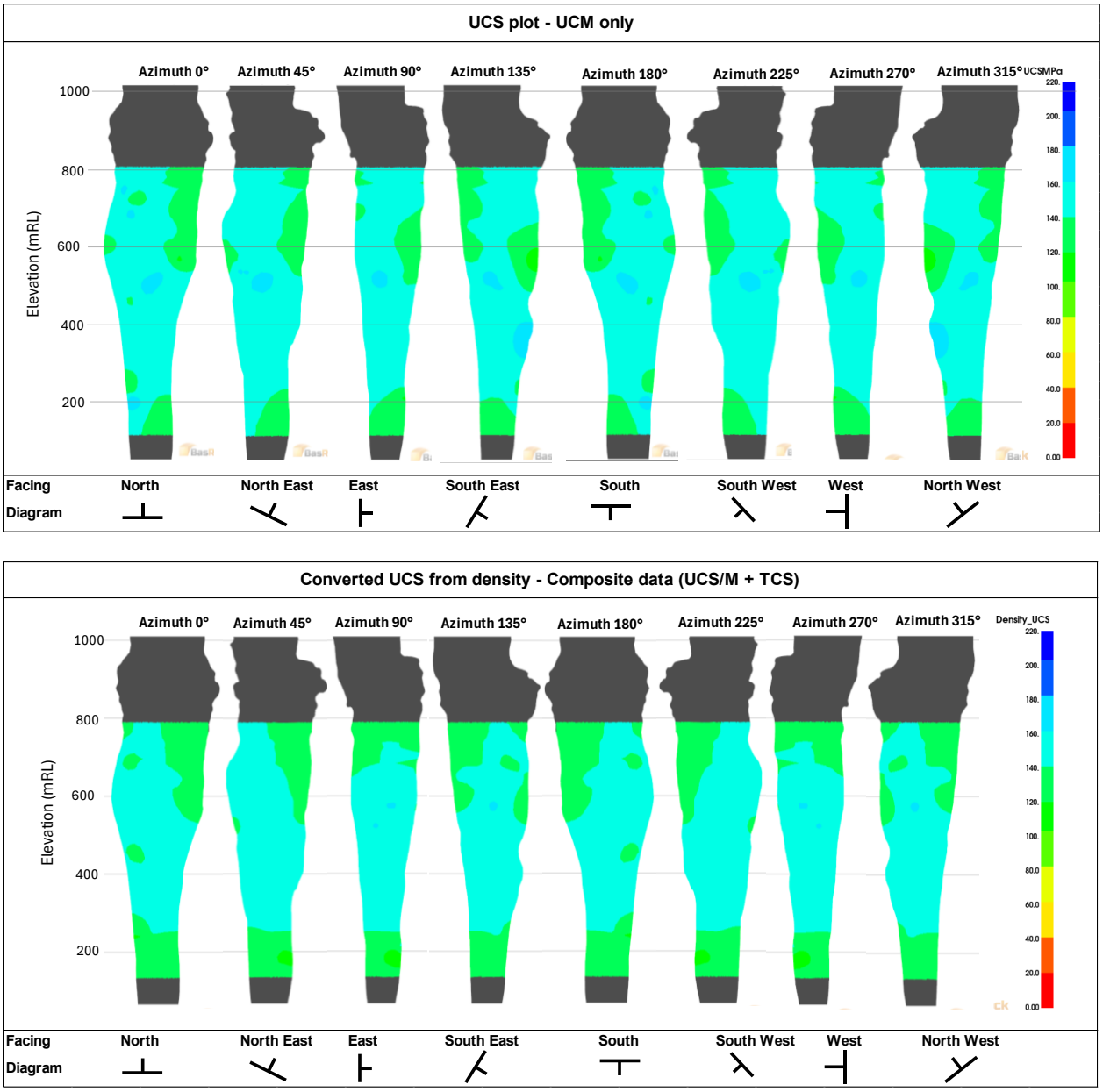
Source: Middindi (2025)

Figure 16-12: Strength versus Density Correlations



Source: Middindi (2025)

Figure 16-13: Pipe Section Visualisation of UCS



Source: Middindi (2025)

Table 16-8: Kimberlite and Country Rock strength statistics

Rock Type	Uniaxial Compressive Strength (MPa)					Elastic Modulus (GPa)			Poisson's Ratio		
	#	Mean	STD	Median	Density (Mean)	Mean	STD	Median	Mean	STD	Median
KAL_BEDS	45	96.63	49.20	102.12	2.42	36.67	17.00	38.10	0.20	0.05	0.20
W_BASALT	19	54.07	24.01	45.88	2.47	17.06	8.77	13.80	0.18	0.05	0.20
UW_BASLT	52	85.70	22.72	86.54	2.63	34.66	8.19	34.50	0.21	0.02	0.21
NTANE_SS	108	29.13	10.31	26.69	2.03	11.39	3.65	10.70	0.20	0.03	0.19
MOS_INT1	24	47.03	13.25	50.33	2.24	18.17	6.80	17.95	0.18	0.04	0.18
MOS_INT2	36	51.95	35.92	40.32	2.32	19.03	14.76	13.25	0.18	0.06	0.18
MOS_INT3	61	40.77	17.52	38.39	2.26	10.77	5.23	10.30	0.20	0.04	0.21
RED_MS1	1	48.25	-	48.25	2.29	27.10	-	27.10	0.19	-	0.19
RED_MS2	12	29.57	11.73	29.49	2.40	7.17	4.22	6.29	0.19	0.06	0.19
TLHAB_MS	84	82.80	24.45	80.34	2.30	18.25	5.95	16.90	0.16	0.03	0.16
TLASS_ST	18	43.29	30.55	34.76	2.28	12.69	6.01	11.25	0.20	0.04	0.19
TLASTMS1	-	-	-	-	-	-	-	-	-	-	-
TLASTMS2	10	85.03	16.18	85.68	2.21	19.50	9.14	17.65	0.17	0.02	0.17
TLA_ARK1	1	27.44	-	27.44	2.49	11.50	-	11.50	0.13	-	0.13
TLA_ARK2	9	52.73	30.16	40.56	2.41	14.85	7.14	14.30	0.19	0.02	0.19
TLA_CMS1	2	44.44	10.96	44.44	2.25	11.34	5.60	11.34	0.21	0.03	0.21
TLA_CMS2	41	63.55	36.01	46.78	2.32	14.37	7.91	13.80	0.15	0.05	0.16
TLA_CMS3	9	84.14	27.02	89.06	2.31	21.39	3.90	22.10	0.17	0.02	0.17
TLA_CMS4	6	87.17	34.40	93.11	2.27	17.40	4.18	18.20	0.16	0.03	0.16
TLA_INTE	-	-	-	-	-	-	-	-	-	-	-
DOL_S	5	209.34	48.07	199.57	2.97	81.54	3.74	82.50	0.27	0.02	0.27
GRA_BASE	60	124.63	68.81	137.64	2.56	51.68	27.78	64.25	0.20	0.05	0.21

Rock Type	Uniaxial Compressive Strength (MPa)					Elastic Modulus (GPa)			Poisson's Ratio		
	#	Mean	STD	Median	Density (Mean)	Mean	STD	Median	Mean	STD	Median
KIM_CEN	5	119.93	31.96	114.27	2.76	44.20	6.48	45.00	0.28	0.03	0.28
KIM_NTH	2	70.46	6.88	70.46	2.60	20.40	0.99	20.40	0.23	0.01	0.23
KIM_STH	78	143.52	35.75	141.95	2.97	69.10	19.88	71.65	0.24	0.05	0.24
KIMB1	9	106.36	17.49	110.38	2.83	45.18	8.15	45.60	0.22	0.04	0.21
KIMB2	25	163.77	31.74	167.21	3.05	85.18	12.27	85.60	0.26	0.04	0.27
KIMB3	7	132.47	29.94	142.99	2.81	53.76	14.20	47.10	0.28	0.06	0.28
KIMB4	34	140.44	33.87	137.61	2.96	65.13	17.02	69.90	0.23	0.04	0.23
KIMB4a	2	151.80	67.18	151.80	3.08	89.70	25.88	89.70	0.23	0.05	0.23
KIMB5	1	137.43	-	137.43	3.03	83.30	-	83.30	0.25	-	0.25

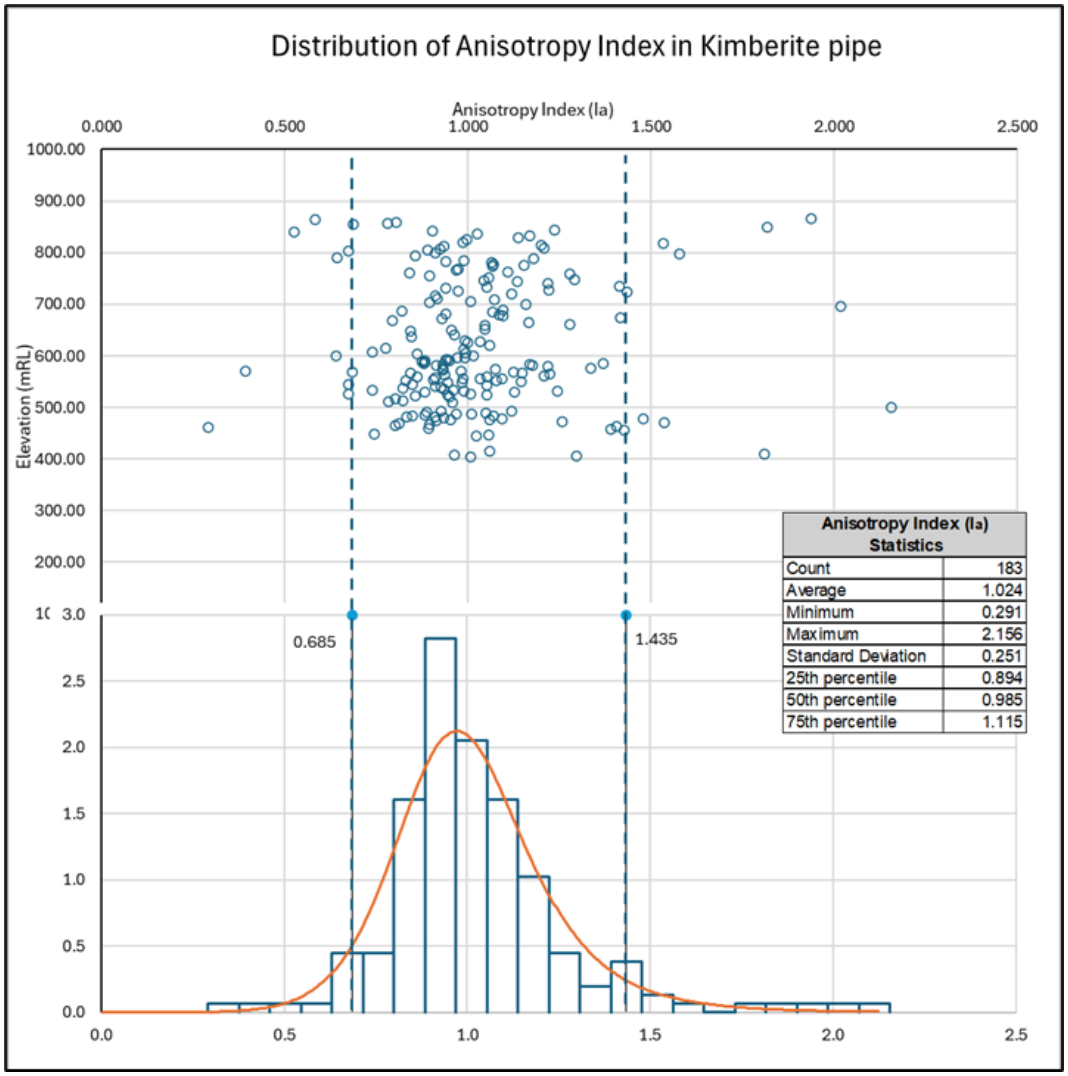
Note:
KIMB2 = M/PK(S); KIMB4 = EM/PK(S)

Source: Middindi (2025)

16.3.3.3 Kimberlite Anisotropy

Analysis of 186 paired diametral and axial point load tests indicated negligible strength anisotropy within the Kimberlite, with anisotropy indices (I_a) demonstrating a mean value of 1.024 (range from 0.290 to 2.156) and clustering predominantly around 1.0. The consistent I_a distribution across all elevations (Figure 16-14) confirms isotropic behaviour, suggesting both homogeneous mineral distribution and the absence of dominant fabric orientation at the tested scale. Although isolated samples exhibited moderate anisotropy, the overall findings validate the use of direction-independent strength parameters for geotechnical modelling of intact Kimberlite units.

Figure 16-14: Kimberlite Anisotropy Index (Ia) Distribution with Elevation and Density Histogram



Source: Middindi (2025)

16.3.4 In Situ Stresses

An understanding of the in situ stress regime is fundamental to the design of stable underground excavations, particularly for the deep, mass-mining operations planned for the UGP. The stress data incorporated is derived from a site investigation campaign conducted by Sigra™ (Pty) Ltd. The investigation utilised the overcore technique with the Sigra™ In Situ Stress Test (IST) tool in borehole KGR_GT_DD011 (hole dip subvertical). The method is two-dimensional and measures the effective principal stresses normal to the drillhole axis. A total of six successful tests were completed across three nominal depth horizons (200 m, 400 m, and 600 m) within the kimberlite orebody.

The averaged results for the principal effective stresses at each depth horizon are summarised in Table 16-9. Given the limitations of the technique, it was necessary to assume the major principal stress is equal to the maximum horizontal principal stress. The data indicates a high horizontal stress regime, where the magnitudes of the horizontal stresses are significantly greater than the vertical (gravitational) stress. The results consistently show that the major principal horizontal stress (σH) is oriented on an East-Northeast to West-Southwest trend, with an average bearing of approximately 062° from magnetic North. The average horizontal-to-vertical stress ratio (k_{avg}) ranges from 1.9 to 2.1, confirming a stress regime dominated by horizontal stresses. The Sigra™ results have been adopted in cave performance numerical simulations.

Table 16-9: Summary of In Situ Stress Measurement Results (Sigra™ Pty Ltd., 2019)

Test Depth Horizon (m)	Major Horizontal Stress, σH (MPa)	Orientation of σH (Bearing from Mag. N)	Minor Horizontal Stress, σH (MPa)	Vertical Stress, σv (MPa)	Average Stress Ratio, k_{avg}
200	11.3	063°	8.6	5.9	1.9
400	27.4	066°	23.2	12.8	2.1
600	33.3	056°	29.4	17.8	1.9

Source: Sigra™ (2019)

16.3.5 Structural Domains

16.3.5.1 Regional Structure

The detailed regional structural footprint is addressed in the geological section of this submission and is hence not repeated here. The defining regional structures is a NW-SE trending fracture corridor that terminates on the pipe boundary. The corridor is a known conduit for water discharge. The as-built and interpreted fault traces for the entire pipe have been incorporated into the mine-scale damage and cave propagation models.

16.3.5.2 Minor Structure

Minor structure (open and closed joints, bedding, layers, veins) were sourced from nineteen boreholes yielding a total pole population for CR and KIMB of 26,696 of which 7,182 pole counts are for KIMB (Table 16-10). ATV orientation data was obtained from 5 boreholes of which 4 traversed KIMB. The large pole count allowed the development of elevation specific orientation analysis within the KIMB segmented into 100 m intervals (Figure 16-15). The following minor structural signature is apparent:

- Below 617 masl, the pipe is defined by a minimum of 3 joint sets but commonly has 4 pervasive sets;
- Above 417 masl, a well-defined low angled (horizontal) set is apparent which may represent banding associated with flow structures;
- Below 417 masl, joint sets are steep (>70 degrees) with a pronounced absence of the horizontal set detected above 417 masl;
- Fractures are spaced on average 0.4 ± 0.36 m apart; and
- Selective bands of high frequency discrete fracture networks exist at four specific regions in the pipe (discussed in earlier sections of this submission).

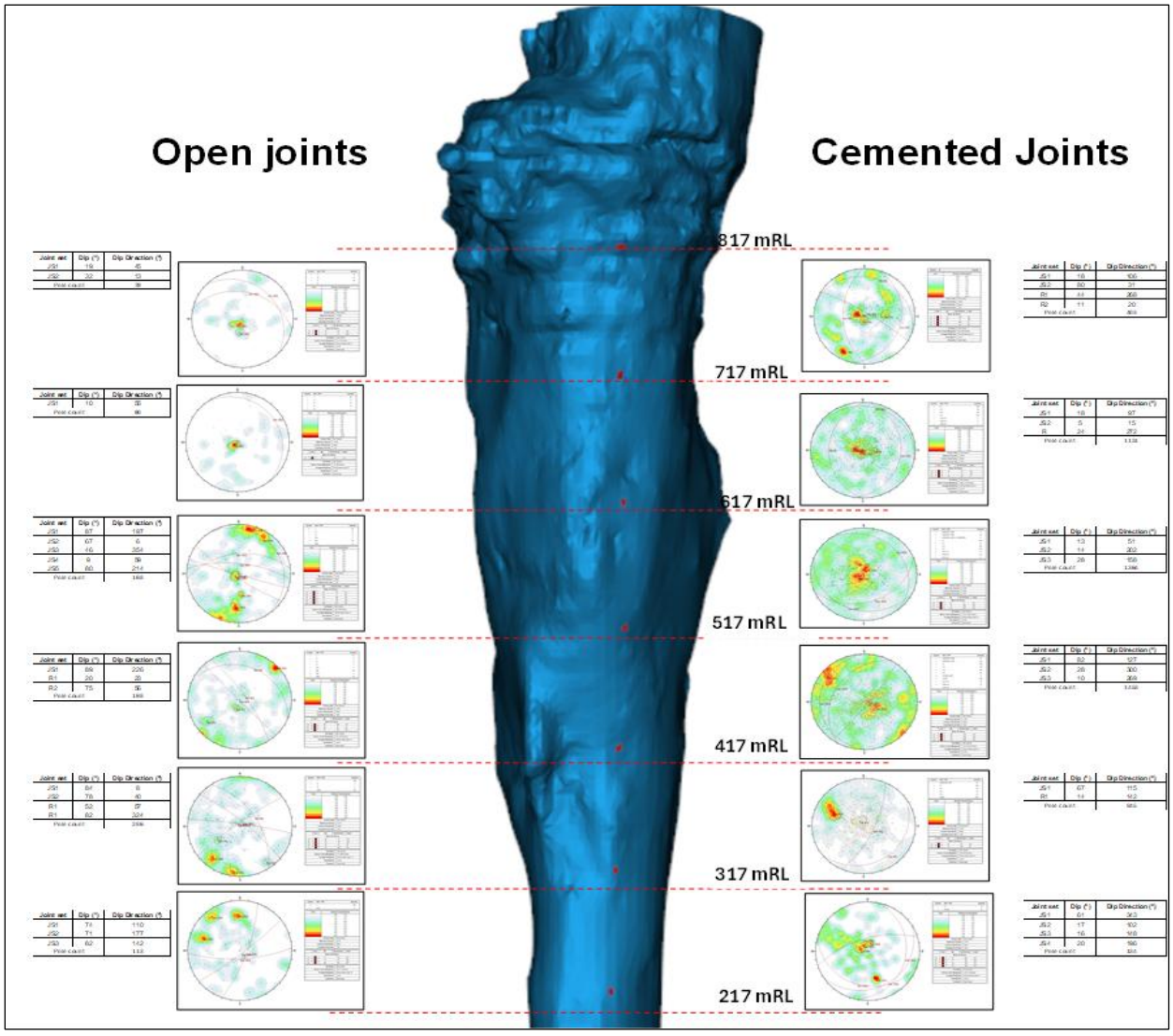
Table 16-10: Source of Minor Structural Data

BH ID	Total -All Data	Kimberlite -ATV	Kimberlite-Structural	Total - Kimberlite Data
CR_GT_DD008	797	-	79	79
GT01a	223	-	53	53
GT02a	1,707	120	193	313
GT03	2642	698	184	882
INFRA_GT_DD003	866	-	196	196
INFRA_GT_DD004	764	-	24	24
KGR_GT_DD002	947	-	357	357
KGR_GT_DD003	1595	-	564	564
KGR_GT_DD004	862	-	210	210
KGR_GT_DD005	910	-	27	27
KGR_GT_DD005A	347	-	317	317
KGR_GT_DD006	1,034	-	399	399
KGR_GT_DD007	1,610	-	727	727
KGR_GT_DD008	964	-	289	289
KGR_GT_DD011	2,077	-	2,077	2,077
REP_003	369	-	80	80

BH ID	Total -All Data	Kimberlite -ATV	Kimberlite-Structural	Total - Kimberlite Data
REP_006B	4,056	182	111	293
REP_009	1,033	-	2	2
REP_012	3,893	182	111	293
Total	**26,696**	**1,182**	**6,000**	**7,182**

Source: Middindi (2025)

Figure 16-15: Microstructure Orientations Related to Down Pipe Elevation



Source: Middindi (2025)

16.3.6 KIMB/CR Contact Conditions

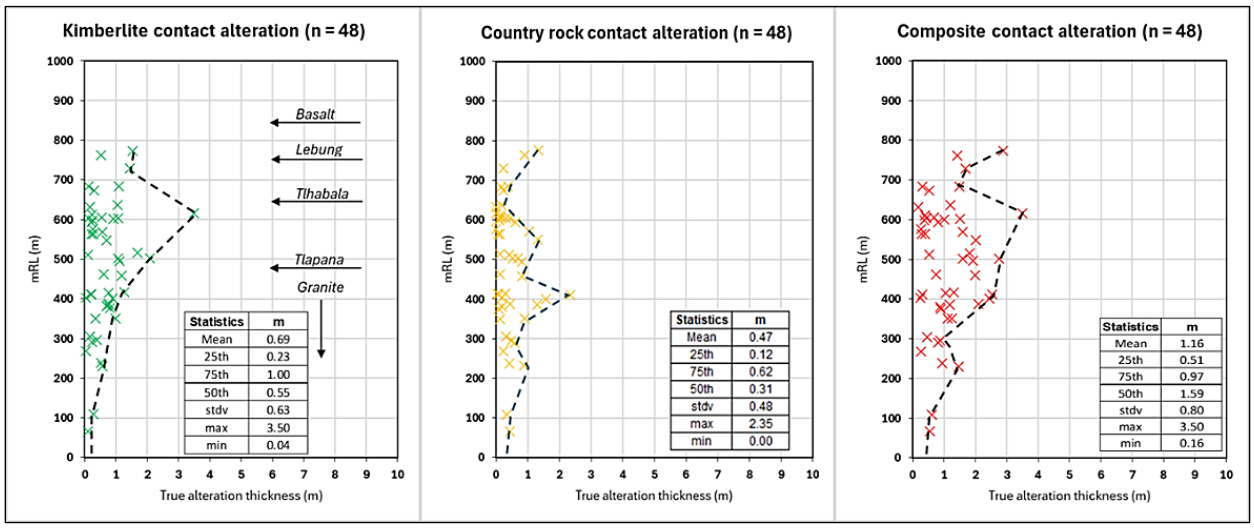
Figure 16-16 presents the distribution of true alteration thickness for composite contact zones (n=48), incorporating both Kimberlite and Country rock alteration data. The alteration thicknesses range from 0.16 m to 3.50 m, with a mean thickness of 1.16 m. Overall, Kimberlite contact zones exhibit greater and more variable alteration thicknesses than Country rock, which may be influenced by differing alteration mechanisms, rock properties, or structural settings at contact boundaries. The mean of all contact types indicates that GSI degrades to 40 (Table 16-11). For numerical simulation purposes, a conservative weakened contact halo of 3.5 m was applied throughout the pipe.

Table 16-11: Kimberlite-Country Rock Contact Rock Mass Classification

RQD (%)	GSI	RMR89	RMR L90	Q - Index	Q' - Index	MRMR
51.73	40.87	45.87	36.19	3.19	7.16	24.96

Source: Middindi (2025)

Figure 16-16: Contact Alteration Thickness Results



Source: Middindi (2025)

16.3.7 Weathering Susceptibility

Hyperspectral imaging provided an assessment of alteration, clay content, and weathering potential across the lithological units. In the country rock sediments, alteration is dominated by saponite and kaolinite/calcite, with lesser illite and chlorite. Alteration in the basalt is also dominated by saponite. The kimberlite shows little alteration overall, but increased saponite and serpentinite are present at the contact between country rock and kimberlite, with minor nontronite, indicating a tendency for expansive clays at these boundaries. The basement granites are dominated by kaolinite in their weathered upper portions and by illite in the fresher lower portions.

The Mosolotsane red mudstones are dominated by saponite, indicating expansive behaviour when wet and rapidly breaks down to residual soil. The Tlapana carbonaceous mudstones are dominated by montmorillonite, which absorbs water and disperses, producing a weak and dispersive material. Both mudstone types also contain variable illite and chlorite. Accelerated wet–dry cycle tests (SRK, 2019), supported by field observations, confirmed the susceptibility of these units to weathering:

- Kimberlite: little to no weathering under repeated cycles, though contacts may degrade due to clay alteration;
- Mosolotsane red mudstones: rapid disintegration within a single wet–dry cycle;
- Tlapana mudstones and coals: moderate resistance, degrading after three to five cycles; and
- Tlhabala unit: relatively competent overall, with only minor samples showing degradation.

In televiewer logs and shaft drilling, complete weathering of Mosolotsane red mudstones was observed, supporting the laboratory findings. Overall, the susceptible mudstones can lose more than 50% of their rock mass strength after short-term exposure to water or atmospheric conditions. The implications of weathering susceptibility are as follows:

- Rock mass strengths for susceptible subdomains are considered representative of in situ conditions if protected from exposure, but significant strength loss is expected if development faces remain unsealed;
- Any underground development in these units should therefore be promptly supported and sealed after exposure to minimise degradation; and
- Long-term monitoring of ground conditions, particularly at kimberlite contacts and in mudstone-hosted areas, is recommended to validate weathering performance under operating conditions.

16.3.8 Cavability Assessments

The primary factors influencing the ability of the South Lobe to free cave is the pipe geometry (shape), IRMR or MRMR, hydraulic radius (HR), degrees of freedom created by fractures, and the ambient stress. These parameters were assessed through the use of established empirical

cave assessment methods (Middindi, 2025) and comprehensive large strain, dilatant, coupled discontinuum and hydro-mechanical models (Beck, 2025).

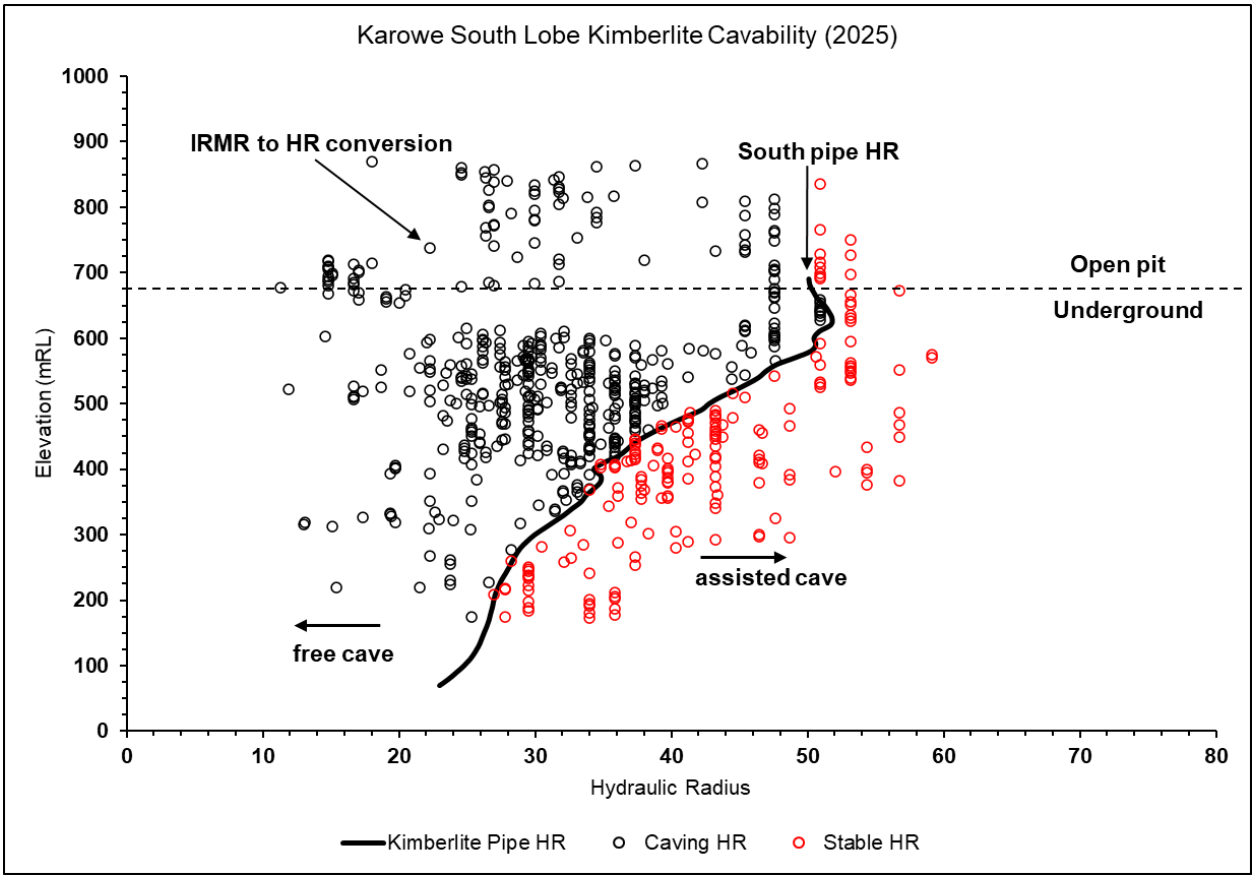
16.3.8.1 Empirical Assessment

The pipe is carrot shaped with an HR of roughly 50 m at the base of the ultimate pit (665 masl) diminishing rapidly to approximately 28 m at the extraction level (310 masl). With similar rock mass quality throughout, an assessment indicated an inability of the kimberlite to free cave on the lower levels, but the likelihood of free caving at the upper levels. The tapered pipe geometry therefore resulted in a change from free (gravity) to assisted caving (blasting). The transition point was derived by simply transposing the measured IRMR (in situ rock mass ratings) onto the actual pipe HR. To allow projection onto the common HR axis, the measured IRMRs were converted into equivalent HR using the Laubscher (1990) relationship between MRMR and HR (Figure 16-17). No adjustments were applied to the IRMR. The transition point was further validated by plotting the actual measured IRMRs per 100 m vertical interval on the Laubscher stability chart (Figure 16-18) and finally visualising the free to assisted cave transition on a series of vertical sections of the pipe (Figure 16-19).

The empirical assessments indicate the following:

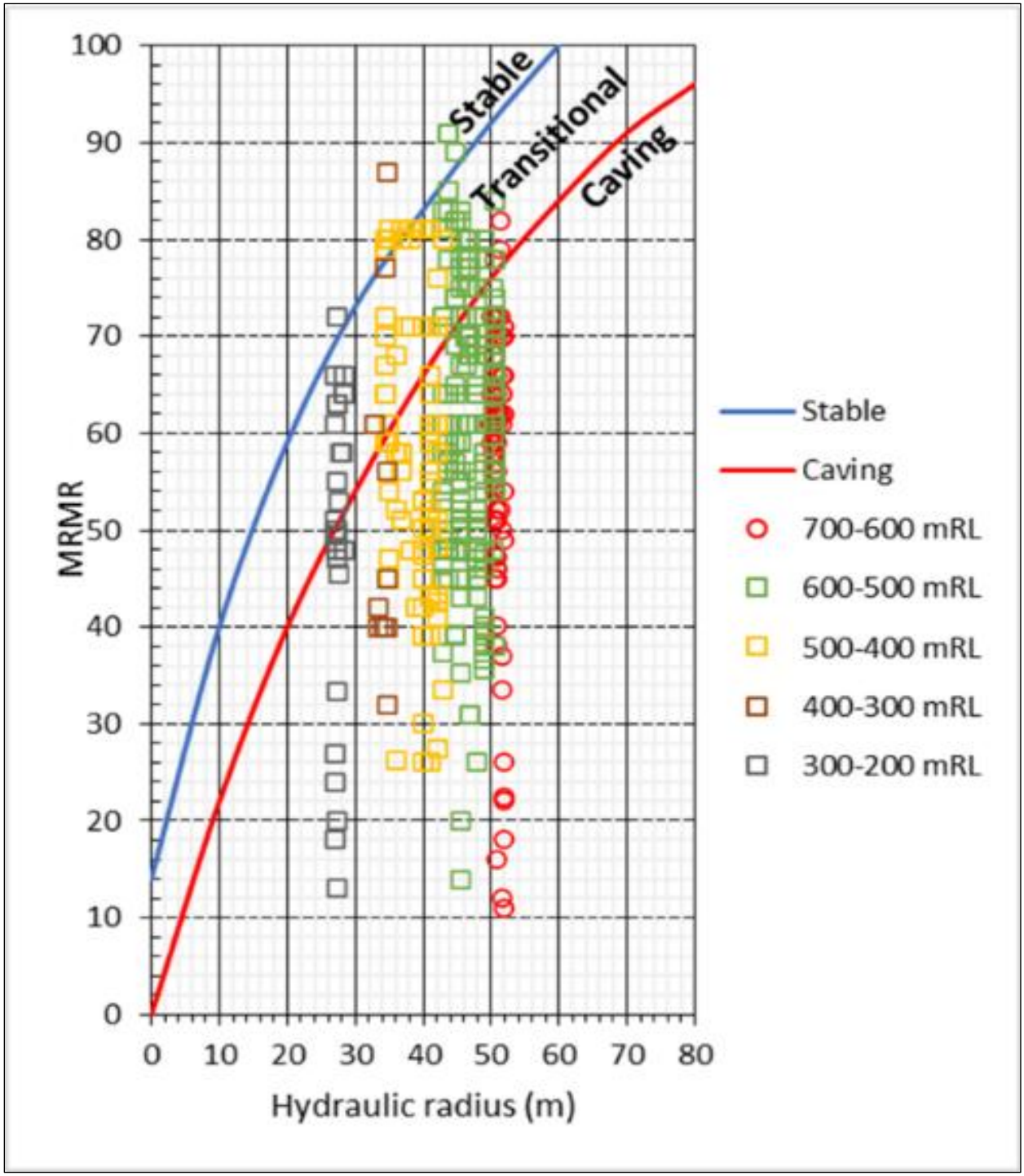
- For all depths, there is a combination of material amenable to free and assisted caving but the proportions that fall within the free versus assisted categories diminishes with depth; and
- The bulk of KIMB rock mass between 665-480 masl is amenable to free cave.

Figure 16-17: Pipe HR Related to IRMR



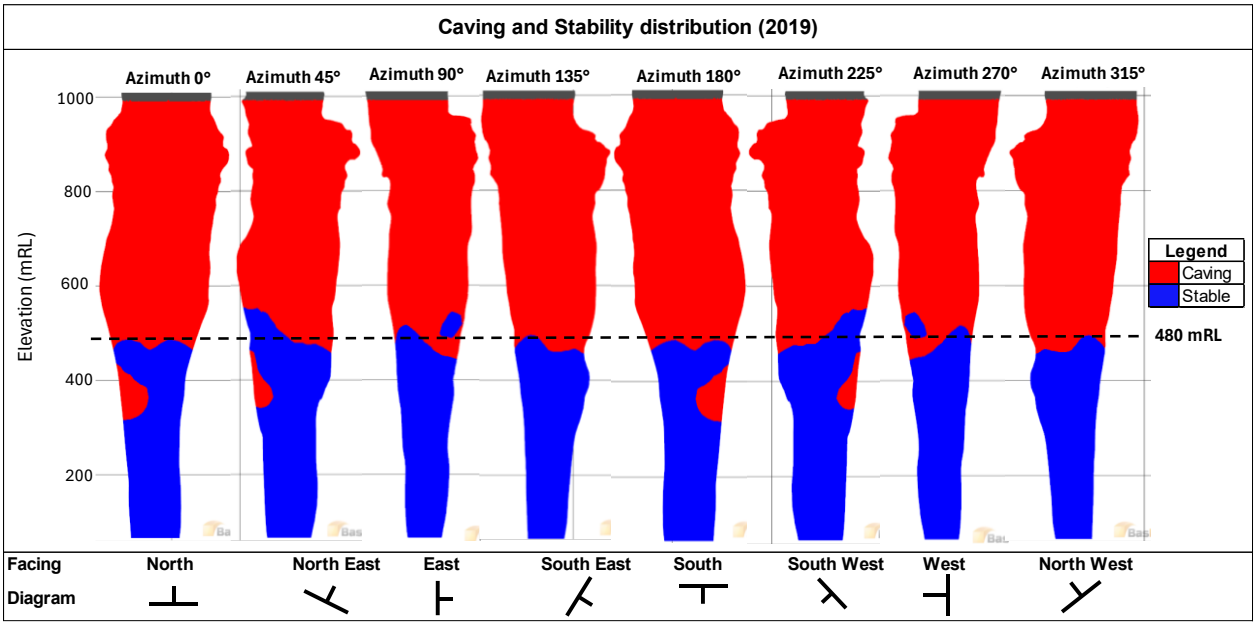
Source: Middindi (2025)

Figure 16-18: IRMR Related to HR for 100 m Level Intervals



Source: Middindi (2025)

Figure 16-19: Free (Red) and Assisted (Blue) Cave Regions for Sections Along the Pipe



Source: Middindi (2025)

16.3.8.2 Degrees of Freedom

Below 417 masl, open joint sets are steep (>70 degrees) with a pronounced absence of the horizontal set detected above 417 masl. The absence of the sub-horizontal set will negatively impact free block movement and the formation of predictable cave back arches which supports the implementation of assisted caving below 480 masl.

16.3.8.3 Numerical Assessment of Assisted to Free Cave Transition

Over several design and modelling iterations, the hybrid Blast Assisted Cave (BAC) plan was shaped to address geotechnical risks as far as practicable. Beck, Lucara and JDS performed work over several months to compile the necessary inputs and the assembled data to formulate a hybrid BAC mining plan, which was then constructed as a coupled LR4-FS4 model.

An initial LR4–FS4 shrinkage-stoping model (Beck, 2025) was developed and run according to the 2023 FS specifications. The model demonstrated that the stope back would undergo uncontrolled collapse preventing continuation of the blasting program as conceived in the 2023 FS at or before approximately 500 masl, though blasting continued past this point according to the schedule. The onset of apparent global instability in the cave back is illustrated in Figure 16-20, which shows extensive damage around the propagating cave and severe degradation of the drill horizon at that level.

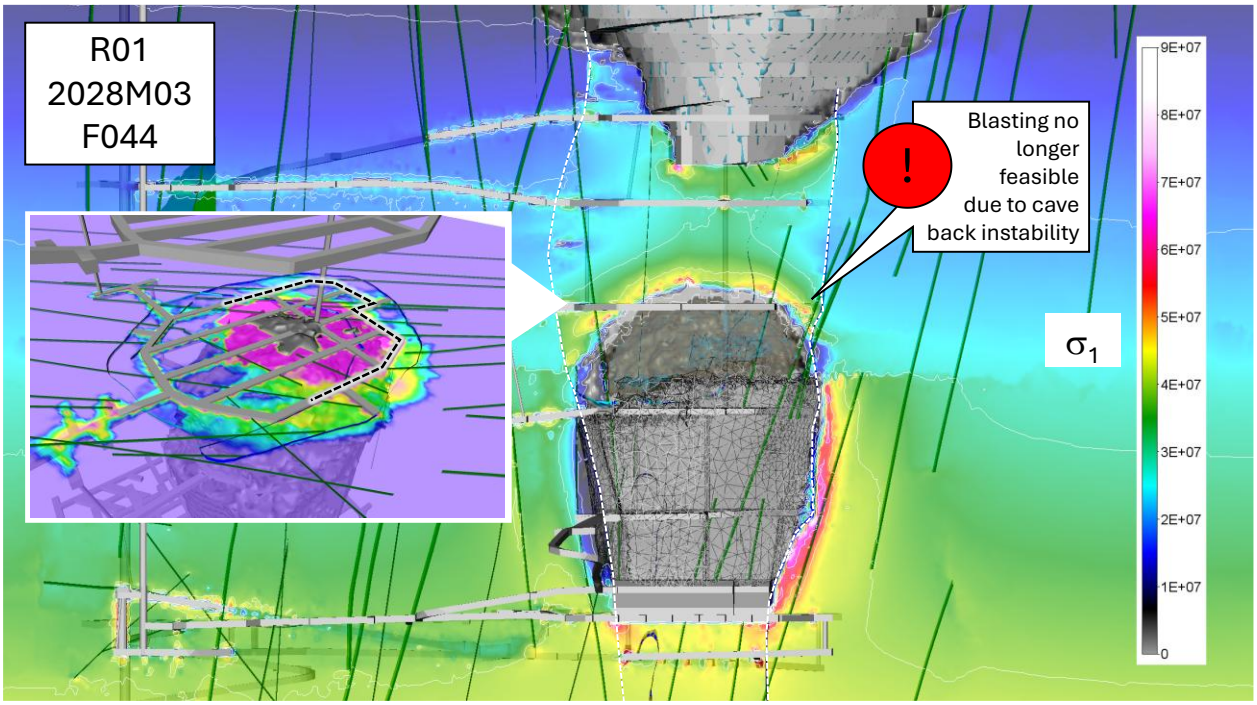
A second model was run using the same development and blasting plan, except blasting ceased at the identified instability point, after which cave propagation was driven solely by draw. The salient findings from this model were:

The model indicated successful cave propagation (Figure 16-21). In the absence of continued blasting, which becomes infeasible from the drill levels once caving initiates, the cave advanced “naturally,” controlled by rock mass structure, geometry, and stress. Draw was more effective to the east, where higher-value EMPK ore predominates, while cave breakthrough to the west was slower in lower-grade areas; and

On this basis, BAC was recommended as the base case, with free cave above ~480 masl. The model also highlighted the conical shape of both the cave and the overlying pit as important controls on the global flow behaviour (funnel flow rather than column flow).

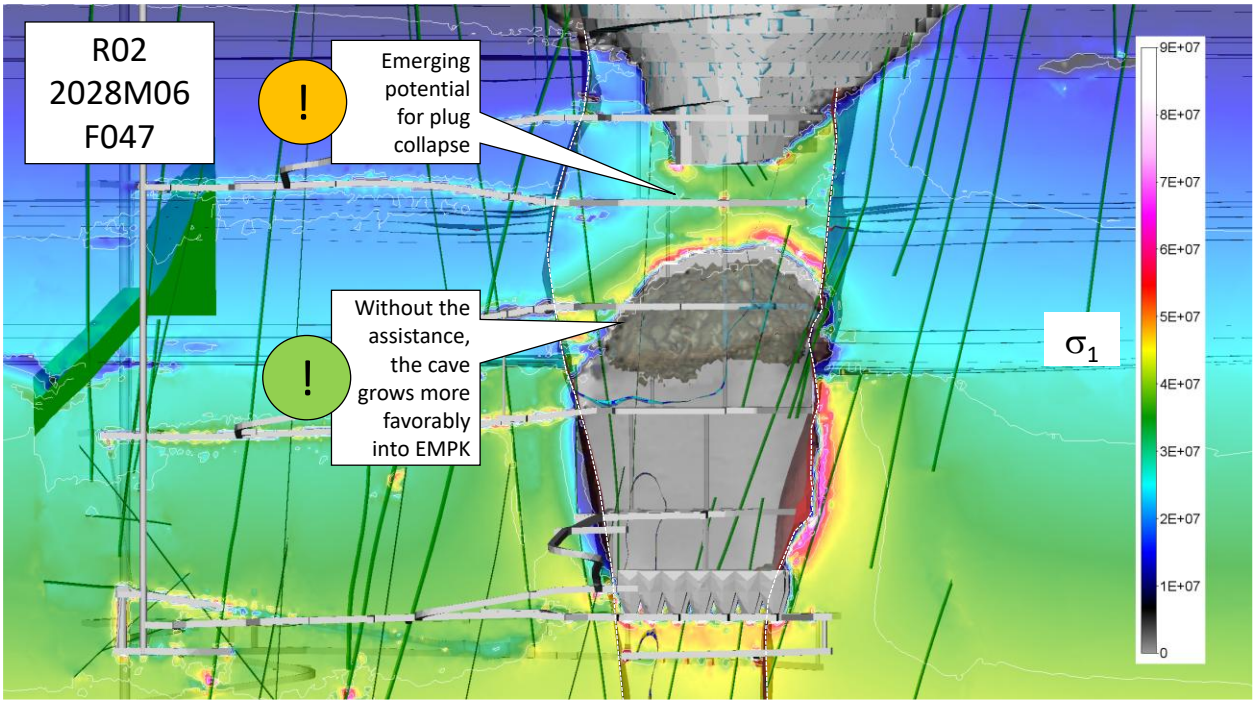
The model also reconfirmed a risk also present in shrinkage stoping, which was potential for collapse of a plug, or rapid caving in the final stages of propagation towards the pit. The main implications would be for management of access if blasting were still taking place, but if a large airgap were present there are ejection hazards in and near the pit, air blast hazard if connected with underground workings, and in the event the sump is undrained, a need to manage the timing of dewatering activities. A comprehensive monitoring plan, and a Cave Management Plan, were therefore developed to monitor and manage these possible outcomes.

Figure 16-20: Cave Back Instability Predicted at 500 masl (Looking North)



Source: Beck Engineering (2025)

Figure 16-21: Early Caving Analysis



Source: Beck Engineering (2025)

The blast assistance in the lower section of the column is needed because of the conical shape of the pipe, and the competent ground conditions; the span is too narrow to cave well, and if it did cave at the narrow span, the high grade near the top of the column would sit outside the reliable draw zones. The blasting targets better caving and better flow for higher recovery.

16.3.9 Caving Strategy

16.3.9.1 Vertical Cave Zoning

For the UGP, the studies support a vertically zoned caving strategy:

- Blast-Assisted Caving (BAC) - Lower zone (≈310–495 masl):
  - The competent, defect-poor rock mass in this zone is not expected to cave naturally under gravity. Controlled drilling and blasting from dedicated horizons (~405 and 470 masl) will create a broken, mobile mass that propagates downward when the undercut at 335 masl is progressed and draw commences. The extraction horizon at 310 masl, designed in a El Teniente layout, will recover the broken ore. Appropriate draw sequencing is essential to prevent stalling or formation of stable voids. The design utilises an advanced undercut sequence, whereby the uphole rings on the undercut are

blasted before the drawbells are created on the extraction level immediately below. This reduces the potential for stress-induced damage of the drawbell pillars during construction.

- Free Caving (BC/FC) - Upper zone (≈500 masl to base of pit at ~665 masl):
  - In this zone the pipe widens, and the rock mass is structurally favourable for caving. Beck's (2025) numerical models show that natural caving can be sustained, though with different propagation styles. This variability underlines the importance of adjusting draw rates and schedules based on real time cave performance instrumentation and monitoring.

16.3.9.2 Elements of BAC

Ultimately the BAC design incorporated both up- and downhole blasthole drilling from two levels, and a sequence which creates an inclined blasting front at 50 degrees (Figure 16-67). The steeper blasting front significantly reduces the risk of fall-off from the undersides of the long-hole rings, by significantly reducing the degree of undercutting of those rings.

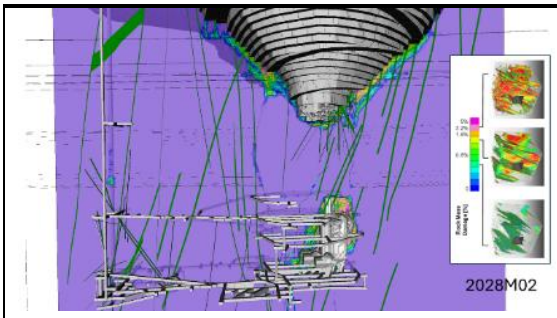
Key elements of the BAC design are as follows:

- The uphole rings on the undercut are first advanced (blasted before draw bells are created);
- Drawbells and drawpoints are constructed in a sequencing lagging behind the blasted advanced undercut;
- The BAC (excluding the undercut) is initiated from two levels. Production drilling occurs initially from the 405 lvl and subsequently the 470 lvl drives;
- Inclined down holes (40 mV) and short inclined upholes (20 mV) are concurrently fired, to achieve an overall 50 degree inclination to the blasting front; and
- Free caving initiates at or around 490 masl.

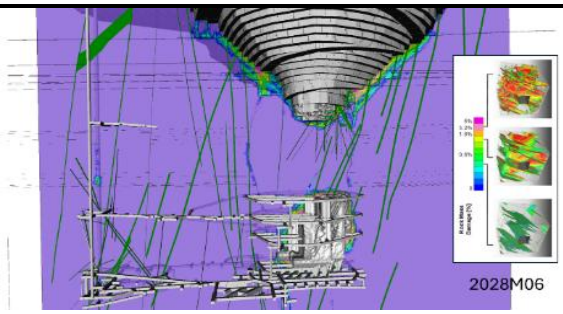
16.3.9.3 Progression of Caving

According to Beck (2025), the propagation of the cave can be broken down into several key milestones, illustrated here using rock mass damage (plastic strain) on the mid-section through the cave (Figure 16-22). Generally, the model indicates that caving initiates relatively rapidly around or just below 500 masl, that cave propagation is favourable, and that it quickly propagates upwards to connect with the pit base.

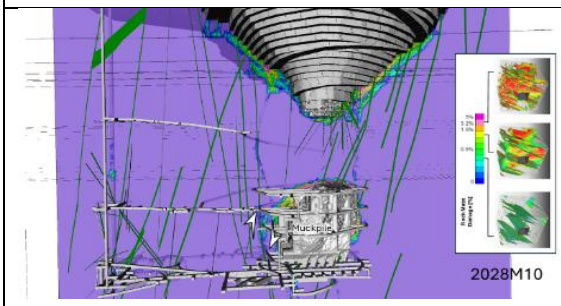
Figure 16-22: Modelled Evolution of BAC and Free Cave (Beck Engineering, 2025)



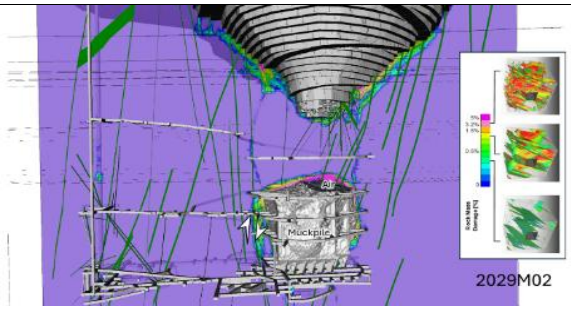
(a) Stable blasting phase, 2028M2



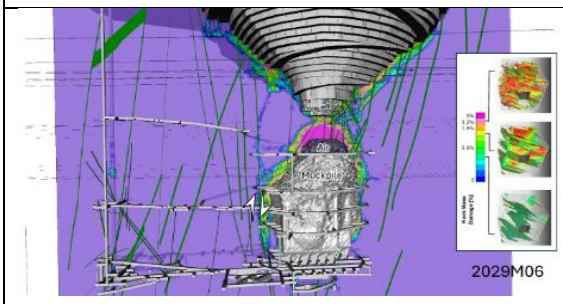
(b) Emerging instability and damage, 2028M6



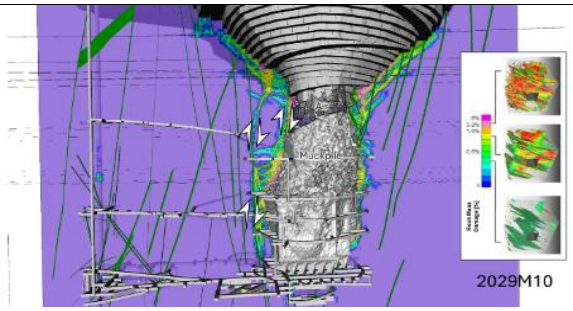
(c) Overbreak beyond drill & blast, 2028M10



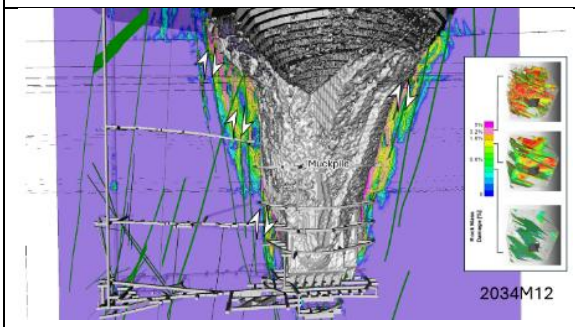
(d) Steady state propagation due to draw, 2029M2



(e) Strong pit cave interaction, 2029M6



(f) Break back onto pit, 2029M10



(g) Mobilisation of the pit, 2034M12

Source: Beck Engineering (2025)

As with all block caves, there are inherent uncertainties and hence risks that need to be managed during detailed design and operations. A variety of strategies are utilised to mitigate and manage these risks, including cave propagation monitoring, draw and air gap management, management of any oversize that reports to drawpoints, and the options of additional blast conditioning should caving stall or be slower than expected, from the available ring drives and the 580 lvl monitoring station. An additional risk reduction strategy for Karowe will be the adoption of wireless blasting technologies (WebGen detonators, etc.), with pre-charged rings “sleeping” and detonated remotely, significantly reducing the need for drill and blast crews to frequently operate on long-hole drilling levels.

The Karowe Cave Management Plan identifies a series of potential risks for each stage of the mining process, and strategies to manage said risks. These include, but are not limited to:

- Localised chimneying;
- Ring fall-off;
- Undercut bridging and drawbell delays;
- Failure to complete the drill and blast operation, leading to more localised or narrow caving; and
- Plug collapse as the cave approaches the pit bottom, with implications for pit bottom activities such as dewatering.

16.3.9.4 Footprint Stability

The geometry of the footprint for the final design is shown with modelled iso-surfaces of significant and moderate damage in Figure 16-23 (Beck, 2025). The model results show that if conditions are as modelled, the footprint will experience a degree of damage that is not catastrophic but will require heavy maintenance and repair and Beck concludes some premature drawpoint loss is likely. The observed damage and its causes in the model include:

- Moderate back instability in the crosscuts in intersections and near the pipe edges, primarily due to the presence of Kimb3, but also stress concentration in the major apex near footprint edges [Figure 16-24(a)];
- Stress and poorer ground at the contacts [Figure 16-24 (b)]; and
- More severe back instability or deep damage where pipe boundaries cross inside the operating footprint [Figure 16-24 (c/d)].

These mechanisms encompass the typical causes of localised instability observed in small kimberlite caves resulting from primarily adverse intersections of rock-mass defects, poorer ground conditions, and interactions with excavations.

Figure 16-24 also illustrates that the model shows all pillar cores remain intact, with only very few suffering moderate damage. The vast majority remain undamaged, meaning the predicted instabilities are limited to controlling surface deformation in the drifts (including backs), which are

normally controlled with more intensive support, or support repairs. The image also shows a stable primary tip and crusher.

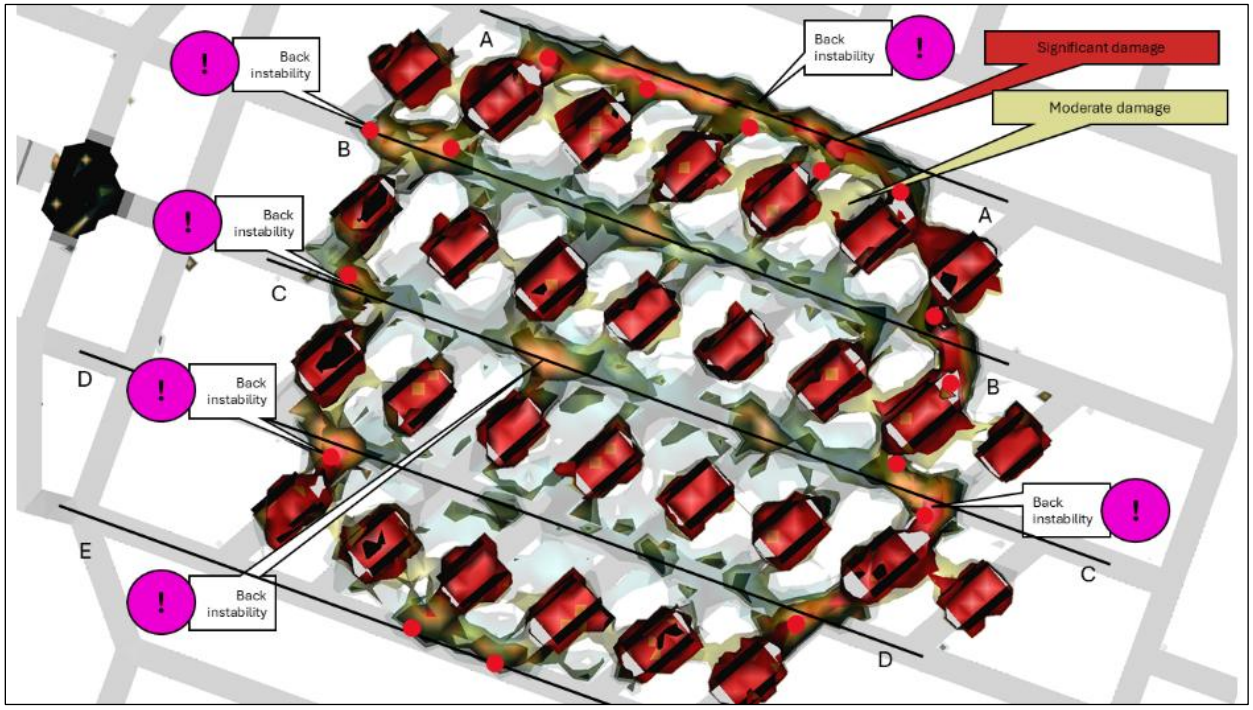
A key challenge for Karowe is that these problems may lead to increased costs and delays, but spatial distribution of these risk factors remains uncertain. The most significant concern is that clusters of drawpoints could be affected simultaneously by poor ground, reducing the availability of a crosscut or limiting draw capacity in that sector. The main risk zones are:

- The northern most crosscut;
- Part of the southern crosscut; and
- The eastern and western contacts.

The current lack of a detailed understanding of ground conditions on the extraction level is, as noted above, a risk. For example, at the time of preparation of this FS the eastern cave edge is only sparsely drilled and there is significant geological complexity in the NW corner that is not fully explained. The pipe boundaries may be differently located or thicker, and there could be non-kimberlite intrusions or pipe contaminants, which is a risk at this stage.

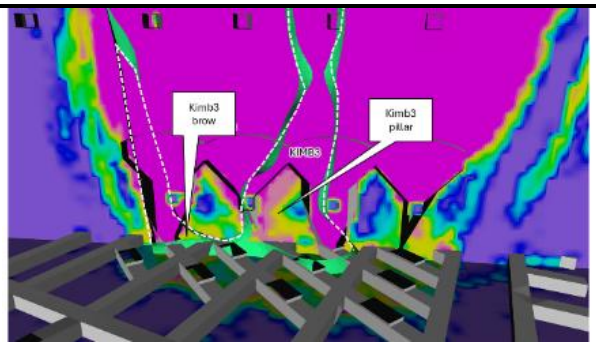
To mitigate the risks arising from uncertainty about the environment, a Cave Management Plan (CMP) has been prepared which proposes a staged development sequence that will allow early identification of emerging problems, updating of geotechnical maps to support cave management, and targeted revisions to ground support (Beck, 2025). Support details are provided below, but this includes steel sets, cables, straps, and heavy fibrecrete application in every drawpoint, and cable anchors throughout the extraction level, at intersections, and at the brows of drawbells. With an adequate contingency budget for heavy support, available heavy support elements and a design for heavy sets if needed, most foreseeable issues should be manageable.

Figure 16-23: Rock Mass Damage on the Extraction Level, End of Mining

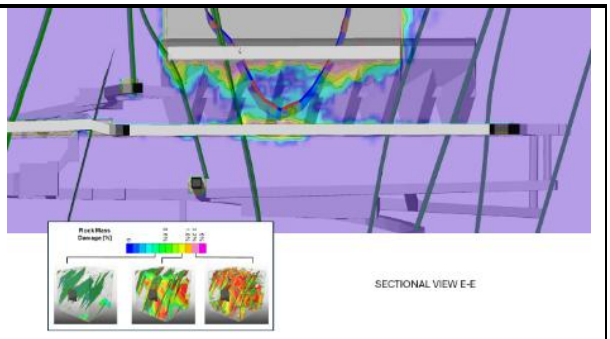


Source: Beck Engineering (2025)

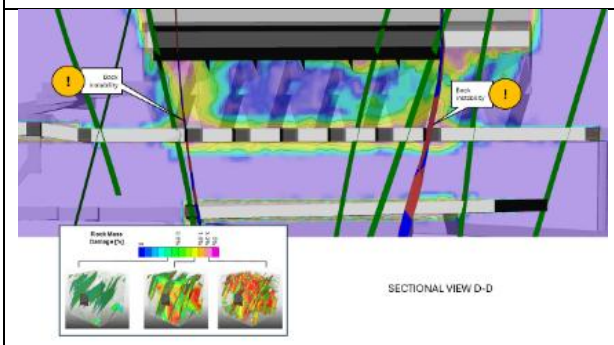
Figure 16-24: Rock Mass Damage Visualised in Sections, End of Mining



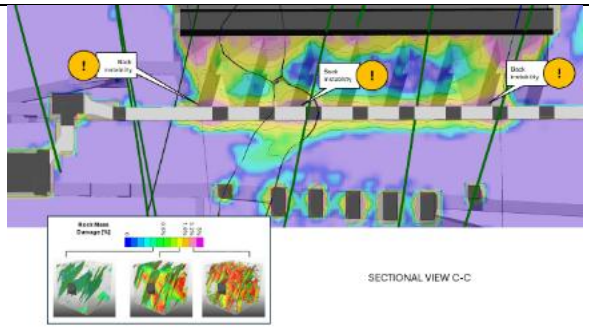
(a) Rock mass damage in a vertical section through the western edge of the footprint



(b) Rock mass damage on Sectional View E-E (Fig 16 24)



(c) Rock mass damage on Sectional View D-D (Fig 16 24)



(d) Rock mass damage on Sectional View C-C (Fig 16 24)

Source: Beck Engineering (2025)

16.3.9.5 Overarching Risk Management Strategies

Hydrogeological and operational controls are integral to the design. Flood control levels, drainage, and pumped sumps are required to manage water inflow. Ground support standards follow the GCMP, with systematic reinforcement of extraction and access drifts, and temporary support at drawpoints during development. Draw control and muck pile management must be carefully implemented to prevent overstressing of pillars and to align cave advance with safe production targets.

In summary, the empirical analysis and advanced modelling both support a dual-regime caving method: Blast-Assisted Caving in the competent lower column and Block/Free Caving in the wider upper column. Beck's modelled scenarios highlighted that while cavability is achievable, propagation style and timing may differ, thus requiring continuous monitoring and adaptive mine management to ensure reliable cave performance.

The UGP monitoring program combines continuous and campaign measurements to track cave propagation, deformation, hydrogeology, seismicity and muck flow. Key instruments include

smart markers, slough metres, micro-seismic monitoring, multi-point and tape extensometers in extraction drifts and pillars, periodic LiDAR/drone photogrammetry and targeted surface scans as the cave approaches the pit, and 3D muck pile/subsidence mapping (satellite/ photogrammetry). A vibrating-wire piezometer network (with standpipes as backup) and flow gauges on drains and sumps will manage pore-pressure and dewatering performance during BAC and BC stages. A micro-seismic array plus blast/vibration monitors will track fracturing and blast effectiveness, while drawpoint cameras, production logging and regular sieve/crusher feed checks provide direct fragmentation and flowability metrics.

All instrumentation should be integrated into a central data acquisition and dashboard system with automated TARPs and predefined responses. Trigger inputs include pore-pressure rises, extensometer trends, micro-seismic rate/energy changes and drawpoint/choke indicators; responses link to operational actions and model recalibration. Governance assigns the Geotechnical Engineer and Hydrogeologist responsibility for interpretation and TARP decisions, with routine model updates and scheduled external reviews to ensure adaptive, evidence-based control of the caving sequence.

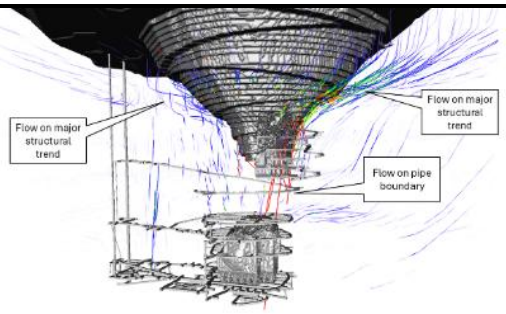
16.3.10 Water Flow into the Cave

The coupled model (Beck, 2025) simulates flow through the cave, but only at coarse resolution, limited by the availability of input data. Images from the model showing flow paths are shown in Figure 16-25, with phreatic surfaces over time in Figure 16-46. Although not definitive, the modelling supports the expected behaviour: the pit geometry, crater shape, fractured zone, and water-bearing structures, combined with near-surface runoff during rainfall events, act together to channel water toward the cave. Water is likely to localise initially along the cave margins but, over time, can migrate into the high-flow pathway as fine, saturated material is drawn into the active flow zone, reducing drainage efficiency and increasing wetting.

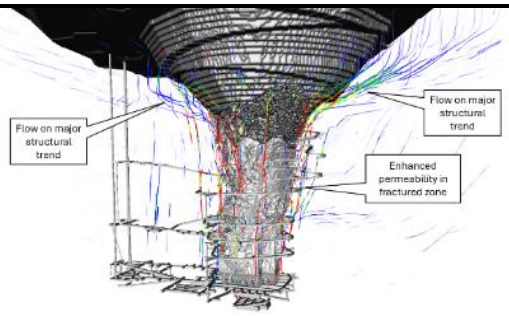
To partially mitigate this principal hazard, the operation will need to maintain effective bunding to limit near-surface water ingress into the cave and implement a mudrush risk management system that explicitly accounts for these higher-consequence scenarios in addition to routine operational risks (see Section 16.7.7.2).

Section 16.4 Hydrogeology discusses explicit hydrological modelling, estimates of groundwater inflows, etc.

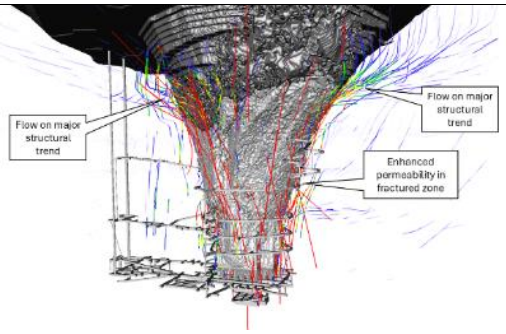
Figure 16-25: Water Inflow in and Around the Cave at Critical Evolutionary Steps



(a) Visualisation of water flows in and around the cave in the model at a late stage of the BAC operations



(b) Visualisation of water flows in and around the cave in the model just after breakthrough



(c) Visualisation of water flows in and around the cave in the model as the cave breaks back into the slopes

Source: Beck Engineering (2025)

16.3.11 Vertical Shafts

The Karowe Diamond Mine underground expansion incorporates two fully concrete-lined shafts, each with a 0.3 m permanent lining designed to ensure long-term stability. Sinking of both shafts has progressed without significant stability issues, with the support system tailored to weak and weathered lithologies, including basalt, red mudstone, Tlapana carbonaceous mudstones, and granite. A special sinking and support sequence was applied in the red mudstone horizon to maintain shaft integrity prior to lining. Long-term stability relies on the permanent concrete linings, groundwater control, and continued monitoring of shaft wall deformation, collar performance, and ground–lining interaction.

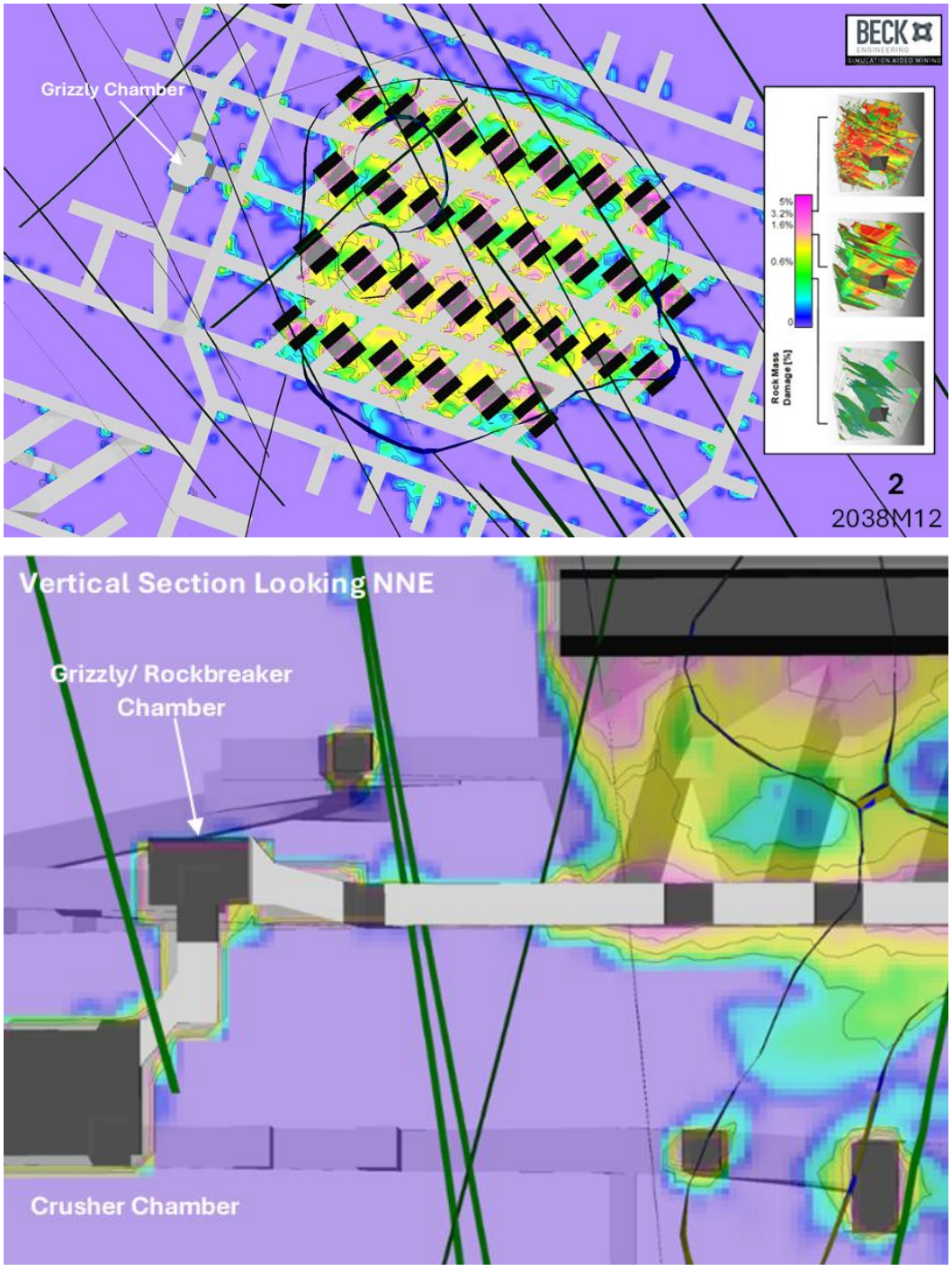
16.3.12 Major Excavations

The majority of the primary infrastructure excavations lie distant from the production areas, and none show any indications of significant rock mass damage during the mine life. The closest excavations to the production areas are the grizzly / rock breaker, ore pass, feed chute, and crusher chamber itself which are 50-65 m west of the closest production drawpoints.

Figure 16-26 shows plan and sectional views through the crusher excavations (Beck, 2025) and indicate only minor rock mass damage around these excavations. It is expected this level of damage can be readily managed with appropriate ground support interventions during excavation.

The crusher chamber and infrastructure excavations at these elevations all lie within the basement granite, and as such ground conditions are expected to be good (see above). A surface diamond drillhole – 2025-027-GGT – was completed in 2025 and was terminated just above the planned crusher chamber (to avoid a surface drillhole within the chamber). This confirmed the good ground conditions in the vicinity of these excavations. Diligence will be performed during development, with appropriate support responses where necessary to ensure long-term stability.

Figure 16-26: Plan and Sectional Views Showing Minor Damage Associated with Crusher Excavations



Source: Beck Engineering (2025)

16.3.13 Ground Stabilisation

Ground stabilisation is informed by a comprehensive ground control management plan (GCMP, Karowe Diamonds, Revision 19). The GCMP covers all elements of ground support which includes inter alia:

- Geotechnical model;
- Ground control districts and geological domains;
- Design acceptance criteria;
- Rock reinforcement and ground support:
- Support specifications;
- Ground support design methodology;
- Temporary support systems;
- Vertical shaft support systems;
- UGP – Free and assisted cave support standards;
- Design philosophy;
- Skin support application protocols;
- Lateral tunnels (level development and drilling horizons);
- Water handling level;
- Extraction level;
- Undercut level;
- Drawpoints and drawbells;
- Large chambers;
- Ground support standards and quality control;
- Stability monitoring, data recording and review; and
- Audits.

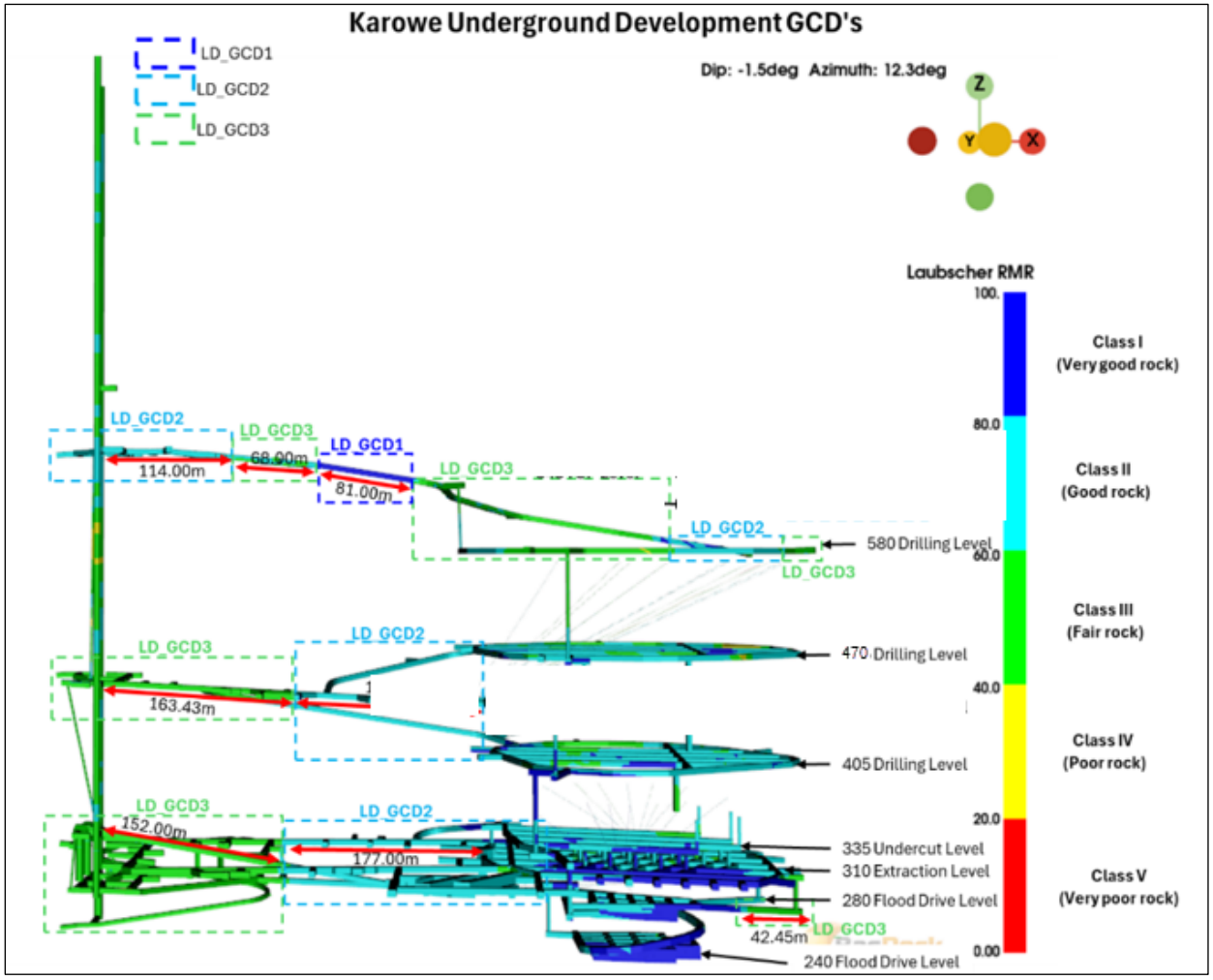
The GCMP (Middindi, 2025) is comprehensive and hence its full contents are not replicated herein. It is a “live” document and must be subject to periodic revisions and updates as the project evolves. The salient features of ground stabilisation are recited in this section. A ground control district (GCD) plan (Figure 16-27) was derived for the underground workings based primarily on the RMRL90 data but for completeness visualisation of RQD, RMR, GSI and Q were also produced. The support classes are aligned with GCDs (Table 16-12) i.e., support class 1 is for GCD 1 and support class 2 for GCD2 and so on. The basic support type and patterns cover all GCDs, but additional support requirements where warranted have been stipulated for the extraction level, undercut level and large excavations. The vertical shaft support regimes have been implemented whilst sinking with all protocols, types and patterns elaborated in the GCMP.

Table 16-12: Support Classes Aligned to Ground Control Districts

Support Classes	GCD	RMRL90
Class 1: Very Good	1	80 to 100
Class 2: Good	2	60 to 80
Class 3: Fair	3	40 to 60
Class 4: Poor	4	20 to 40
Class 5: Very Poor	5	3 to 20

Source: Middindi (2025)

Figure 16-27: Ground Control Districts Aligned to Rock Mass Classifications



Source: Middindi (2025)

The typical ground support types and specifications are referenced in the GCMP. The type, length and pattern changes in accordance with GCD i.e., GCD1 being the least onerous support types, lengths and patterns and GCD 5 being the most intense. The full specifications for ground support in the suite of different excavation types and dimensions are detailed in the GCMP. However, we note the following:

- Laterals (level development) – 2.4 m long resin rebar, mandatory application of mesh and/or shotcrete (thickness increases with increasing GCD) and cable anchors (mainly at intersections);

- Extraction level – in addition to lateral support, steel sets shall be installed at draw points; draw point bullnoses shall be supported by several horizontal rows of tensioned and grouted

steel cable anchors with tendon straps behind the horizontal cables; and shotcrete thickness has been maintained at nominally 150 mm at draw point and apices;

- Undercut level – 1.8 m long split sets are universally used with shotcrete applied only in GCD 4 and 5; and

- Large chambers:

  - Back (hanging wall): Primary support of 3.0 m long rock bolts installed on a 2.4 m by 2.4 m pattern. This is supplemented with secondary support of 6.0 m long cable anchors installed on a denser 3.0 m by 3.0 m pattern;

  - Walls: Support will consist of 2.4 m long rock bolts installed on a 1.6 m by 1.6 m pattern, extending down to a grade line 1.5 m above the footwall;

  - Skin Support: 50 mm thick shotcrete must be applied to all back (hanging wall) and walls to the floor;

  - Ore passes, Silos and Sumps: For excavations such as ore passes, silos, and sumps, where the installation of rock bolts is impractical due to the abrasive nature of material flow, the standard bolting shall be replaced. In these areas, the primary ground support shall consist of a minimum 100 mm thick layer of shotcrete. This shotcrete lining must be applied and allowed to cure before any subsequent drilling operations can commence within the excavation; and

  - Silo Drawpoints: For the drawpoints of the silos (7.6 m diameter), a specific, high-capacity support standard is required. A systematic ring of 6.0 m long cable anchors shall be installed radially around the circumference of the opening. These cable anchors shall be spaced a maximum of 2.0 m apart and positioned 1.0 m back from the final edge of the silo excavation. In conjunction with this reinforcement, the entire excavation surface shall be supported with a minimum 100 mm layer of shotcrete.

16.3.14 Instrumentation and Monitoring

The instrumentation and monitoring (I&M) strategy adopted for the UGP conforms to that typically implemented in cave mining operations. This section focusses on the geotechnical monitoring strategy – hydrological monitoring is discussed under Hydrogeology Analysis in Section 16.4.5 of this Report.

The I&M considers the following overarching factors:

- The scheduled timing of drilling and instrument installation relative to the mine access and cave progression schedule;

- The instrument packages for monitoring both the vertical and horizontal components of cave-front progression for blast assisted and free caving;

- The key geological horizons within the cave and country rock including pipe-country rock contacts, the kimberlite and country rock peripheral to the cave;

- Horizons susceptible to high groundwater discharge; and
- The cave propagation is influenced by both the orebody and country rocks.

The monitoring program covers the following aspects:

- Cave propagation and ground movement monitoring (both vertical and horizontal component and magnitude of movement);
- Country rock displacement monitoring (extent and magnitude of movement); and
- Draw and material flow monitoring.

16.3.14.1 I&M Phases

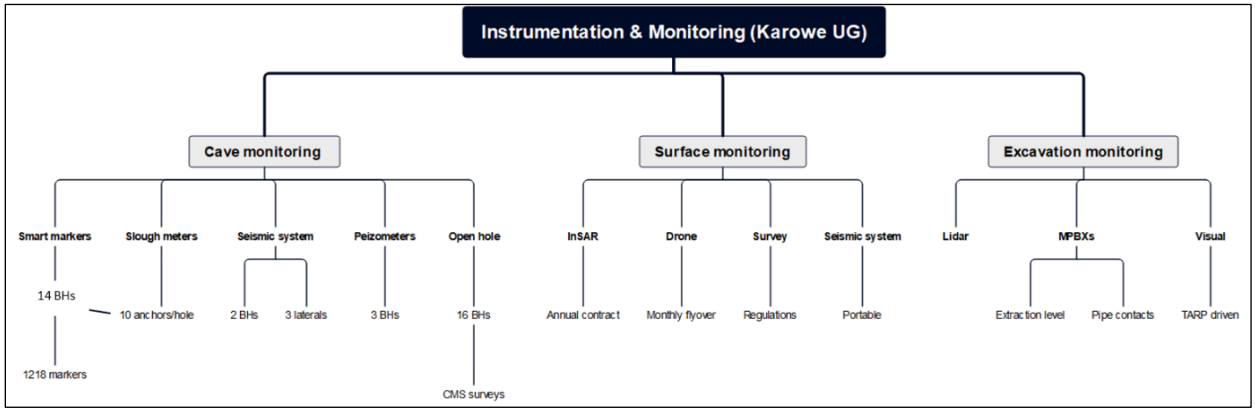
The monitoring program will be carried out in the following phases of which this submission provides the details of Phase 1 and 2:

- Phase 1 – Provisional Design and System Selection: Based on current knowledge of geology and mining, the different alternatives and approaches were evaluated, and an appropriate system recommended. The justified cost for budgetary purposes has been incorporated into the capital and operating expenditure;
- Phase 2 – Final Design: This would take into consideration the final open pit and underground design, updated geology and structures. The detailed costing and manufacturing schedule would be included;
- Phase 3 – Installation and Commissioning phase: This would follow Phase 2. The provisional design will be reviewed and updated if material changes occur in the mine plan. The client, together with the instruments manufacturer, should provide supervision of the drilling program, installation of instruments and commissioning of the systems; and
- Phase 4 – Monitoring and Data Analysis: In this phase, a third party or the manufacturer should provide training of mine personnel to develop the ability to record, analyse and interpret monitoring results.

16.3.14.2 I&M Approach

The I&M programme has been subdivided into three distinct areas (Figure 16-28), but the strategy is considered integrated i.e., the results obtained from I&M must be collectively assessed to inform decisions.

Figure 16-28: Designed Instrumentation and Monitoring Programme, UGP



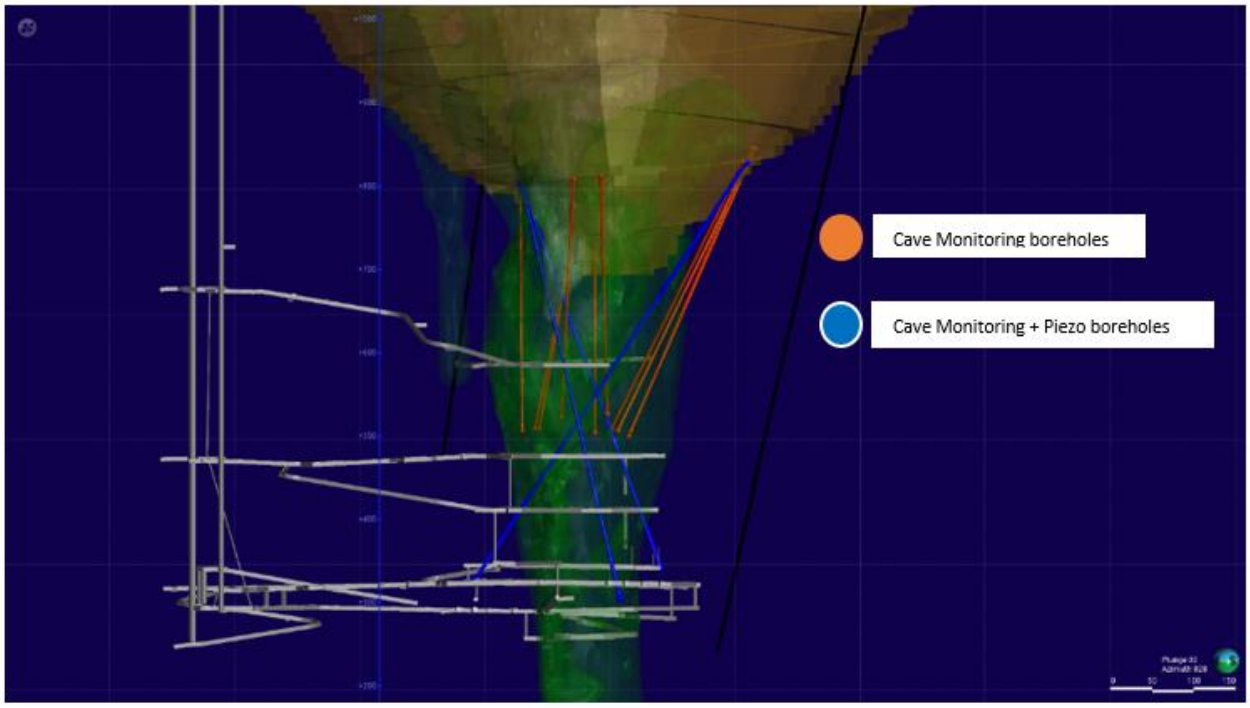
Source: Middindi (2025)

16.3.14.3 Cave Monitoring

The elements of the cave monitoring strategy are listed below:

- Thirteen (13) boreholes will be drilled from the pit benches. The boreholes will provide I&M coverage for the free cave zone above 500 masl. As far as possible, the use of geological infill resource drilling holes will be prioritised (Figure 16-29);
- For the blast assisted cave, strategically positioned uncharged holes will be equipped with smart markers. Installation will predominantly occur from the 470 lvl drilling horizon;
- Geo4Sight Smart markers will be installed in all boreholes averaging about 60 markers per hole. The smart markers will provide three-dimensional tilt measurements which will be used to track cave development and progression;
- The same boreholes will be equipped with Ramjack Smart slough metres which will track rockwall decoupling or sloughing within the drillholes;
- All smart markers and slough metres will eventually be consumed by the cave;
- A microseismic system will be installed underground using seven triaxial geophones. Two geophones each will be located in boreholes INFRA-GT-DD011 and KGR-GT-DD009 with the remaining three installed in short drillholes located from laterals. An additional geophone will be installed approximately 20-30 m below the extraction level to provide data on microseismicity associated with drawbell pillars; and
- Sixteen boreholes will be used for open hole surveys (borehole cameras and cavity monitoring systems).

Figure 16-29: Monitoring Boreholes from Open Pit Benches (Looking North)



Source: Middindi (2025)

16.3.14.4 Surface Monitoring

Surface monitoring is purposed to indicate anomalous cave break backs, tensile crack development and propagation, time to pit break-through and subsidence encroachment onto LOM surface infrastructure (twin vertical shaft system).

The I&M systems will consist of:

- Interferometric Synthetic Aperture Radar (InSAR) – technical and cost proposals have been procured from SkyGeo which covers the following baseline scope: Deformation maps, quality assurance and control of radar shift imaging data, monthly reports, high resolution multi-spectral imagery and a digital model. Annualised costs and support services have been received;

- Drone imaging – drone flyovers will occur at least monthly and increase in frequency to weekly once free cave is initiated. Coverage will include the surface terrain covering the open pit, tailings storage facility, waste management facilities, the shafts and laydown areas. High resolution images together with InSAR and slope stability radar will be used to detect surface fracture location, frequency and progression;

- Survey – routine survey of particularly pit geometrical metrics will continue even after pit closure to quantify the spatial location and magnitude of expected deformations; and

- Mobile surface seismic system – three geophones will be installed on surface and fully coupled with the underground geophones. This will provide a minimised location error array and allow for detection of microseismicity associated with the seismogenic zone of the cave. In addition, the network will be used to assess peak particle velocities emanating from the blast assisted cave.

16.3.14.5 Excavation Deformation Monitoring

An integrated system of light detection and ranging (Lidar) surveys, installation of multi-point borehole extensometers (MPBXs) in select locations, and visual inspections of excavations will provide performance I&M for critical laterals and LOM chambers. A summary of the strategy is outlined below:

- Lidar surveys will focus on deformation monitoring (by assessing progressive shape changes) on the extraction and undercut levels from initiation to depleted serviceability of these levels. Focused and area specific Lidar surveys will occur on ±30 m either side of the transition zone of excavations from country rock into Kimberlite; and

- MPBXs will be installed in 20-100 m long drillholes with two such holes located at the crusher chamber and the remaining six into the interburden between the extraction and undercut levels. Smart MPBXs are typical, easy to procure and install and hence operational needs will drive the installation of these instruments at other locations.

Visual inspections are by far the simplest and most effective method for collecting excavation performance data. Suitably qualified geotechnical specialists will be allocated to UGP to fulfil the following functions (not exhaustive) associated with I&M:

- Face mapping of excavation ends prior to the next blast being executed to record rock quality data, ground stabilisation efficacy and QAQC;

- Perform in situ geotechnical risk assessments and recommend corrective measures;

- Develop, implement and monitor trigger, action, response protocols/plans (TARP);

- Catalogue, interpret and risk categorise data received from the integrated I&M system; and

- Communicate timeously unexpected and high-risk trends and patterns from the I&M system.

16.4 Hydrogeology Analysis

16.4.1 Introduction

Between the release of the 2023 FS report and the effective date of this Report, the following updates have been made:

- The mine plans and schedule for both the OP and UG have been updated;

- The groundwater flow model was updated in 2025 by Aqualogic in Botswana (Aqualogic 2025). The updated model is referred to hereafter as the 2025 Flow Model;
- The P/S sinking was completed, V/S sinking nearly completed, and underground development is underway;
- Inflow rates from underground development were measured; and
- The depressurization target for the OP slope was updated.

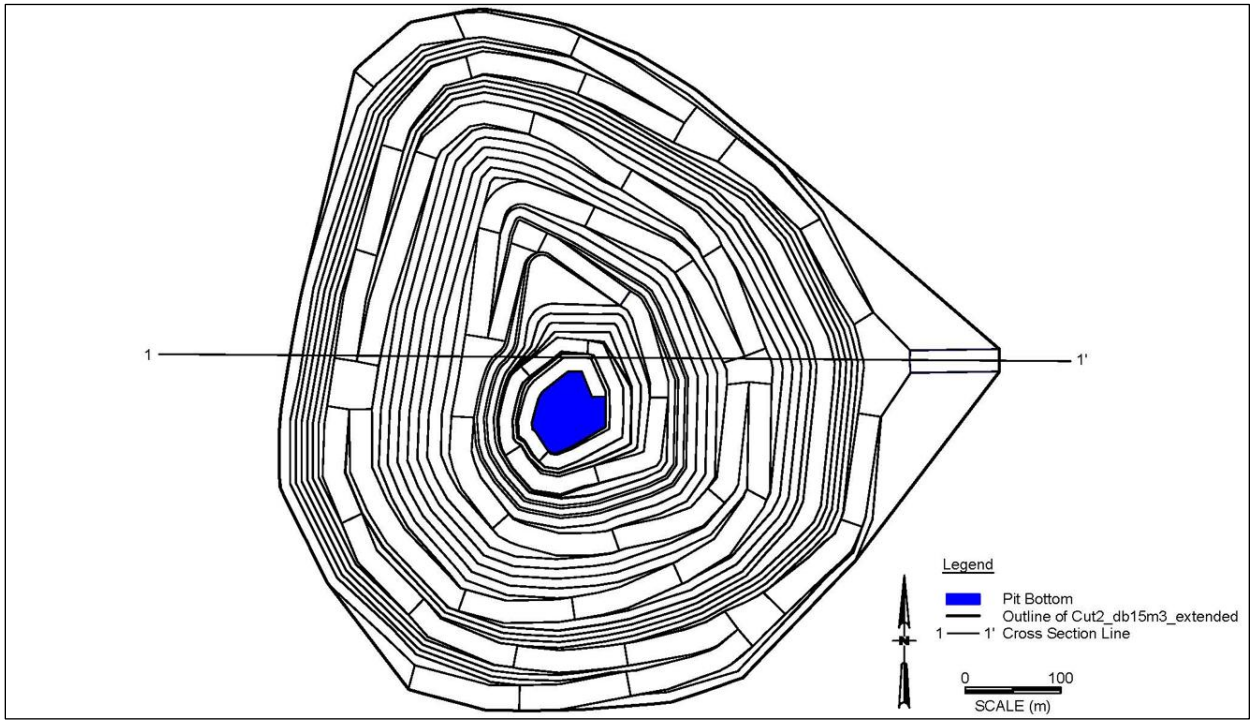
These key updates from the 2023 FS, along with basic hydrogeologic information and updated groundwater flow model predictions, are presented in this section. To differentiate from the 2025 Flow Model by Auqualogic, the flow model that incorporated the mine plan for the 2025 FS is referred as the 2025 FS Model. The 2025 FS Model was used to simulate the updated OP and UG mine plans and predict the pore pressure distributions and inflow over the LOM for the 2025 FS.

16.4.2 Mine Planning and Scheduling

Both the OP and UG mine plans have been updated from the 2023 FS. The OP mining will be completed in April 2026, which is an extension from the date of May 2025 used in the 2023 FS. Figure 16-1 shows the footprint of the ultimate pit of Cut 2. Figure 16-30 shows the footprint of the ultimate pit of Cut 2. Figure 16-31 and Figure 16-32 show cross sections with the pit shell of October 2025 and the ultimate Cut 2 pit shell, respectively. Also shown in Figure 16-31 and Figure 16-32 are the simulated phreatic surfaces from the 2025 Flow Model. The simulated phreatic surfaces will be discussed in Sections 16.4.4.1 and 16.4.7.3. The following observations can be made from Figure 16-31 and Figure 16-32:

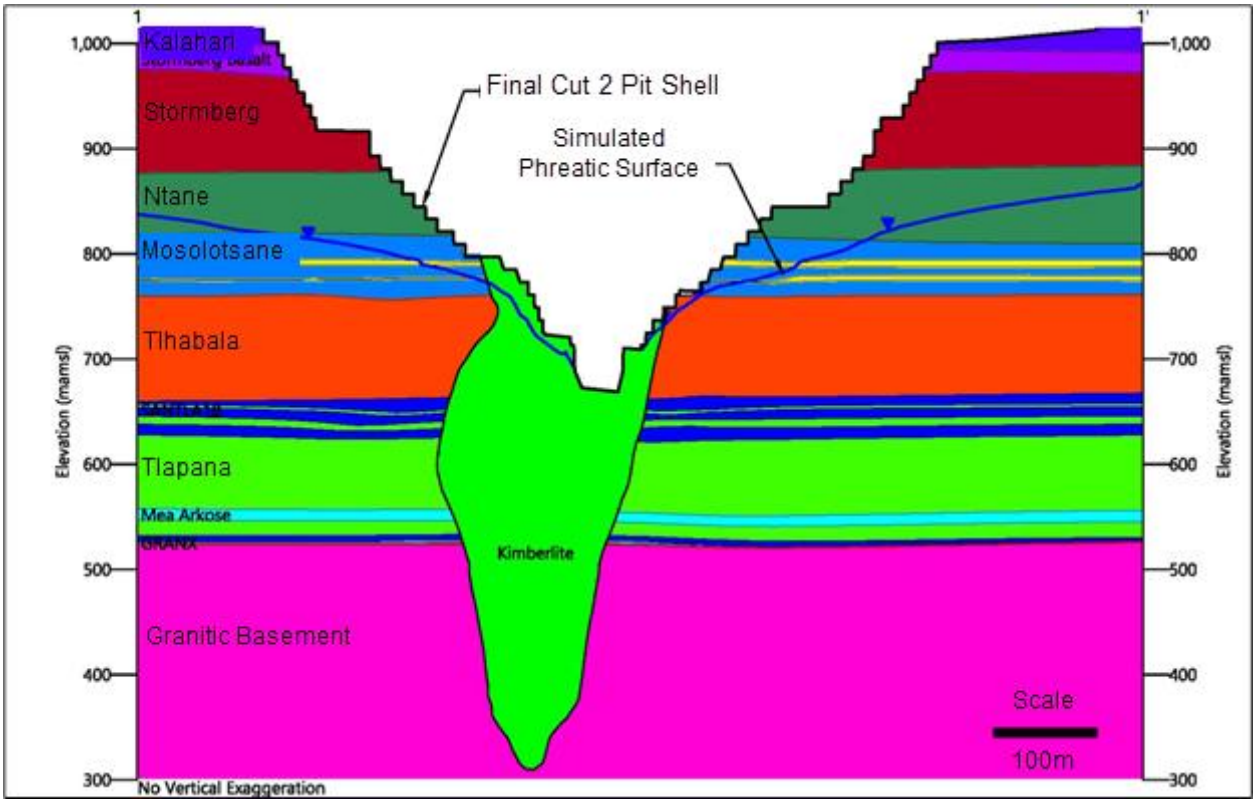
1. The pit bottom elevation as of October 2025 is approximately 707 metres above sea level (mL) and is within the MUDTLH18 mudstone horizon;
2. The final pit bottom elevation is approximately 665 masl and is within the MUDTLH18 mudstone horizon; and
3. As shown in both Figure 16-31 and Figure 16-32, the rock for the deeper portion of the future pit wall is within the kimberlite unit. It is likely that most of the deeper portion of the future pit wall will be within the kimberlite unit.

Figure 16-30: Footprint of Ultimate Pit and Location of Cross Section Line



Source: Lucara (2025)

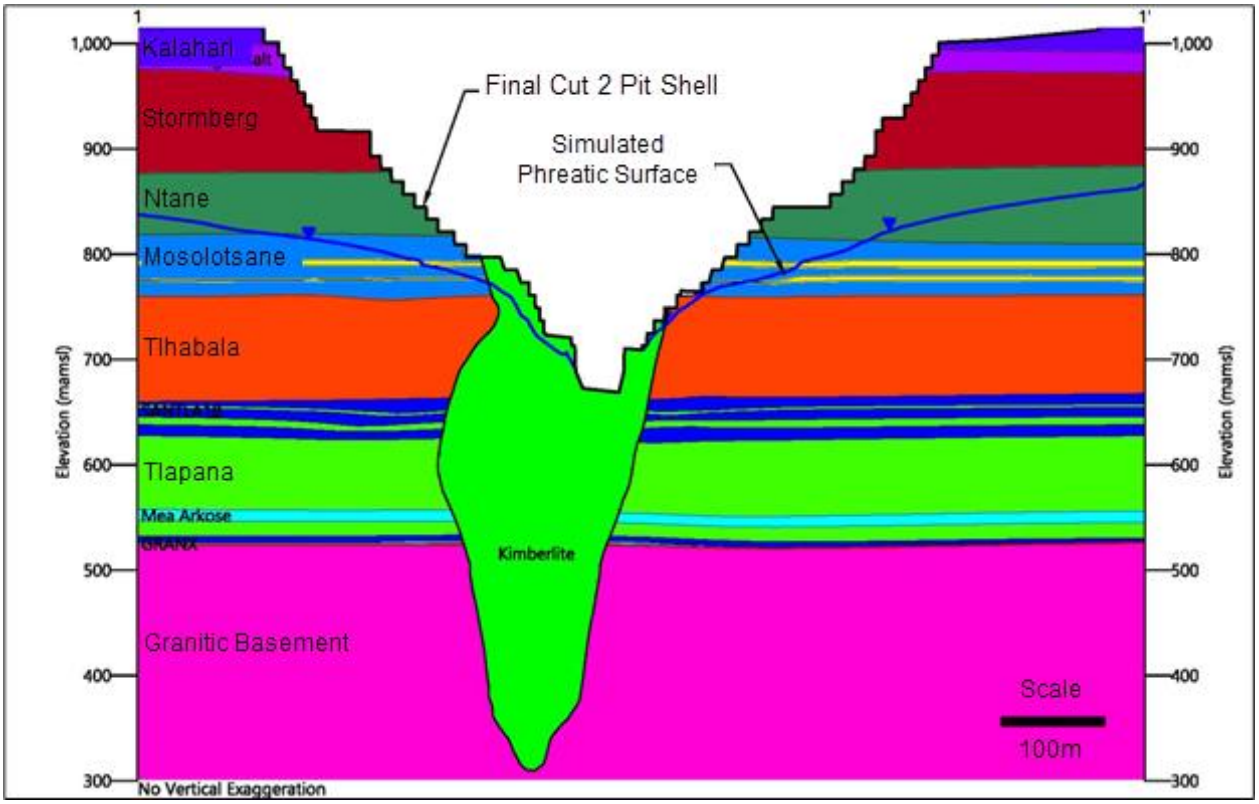
Figure 16-31: East-West Cross Section with October 2025 Pit Shell



Notes:
Vertical and Horizontal scale are equal. Section is looking North

Source: Itasca (2025)

Figure 16-32: East-West Cross Section with Final Cut 2 Pit Shell



Notes:
Vertical and Horizontal scale are equal. Section is looking North

Source: Itasca (2025)

The updated UG mine plans are shown in Figure 16-33 and Figure 16-34. The mining schedule is provided in Section 16.12. The dewatering and drainage system for the UG mining is shown in Figure 16-34. A total of 48 drain holes are planned at four different drain levels to dewater the mining areas. The average length of the holes is 171 m.

Figure 16-33: UG Mine Plan General Layout

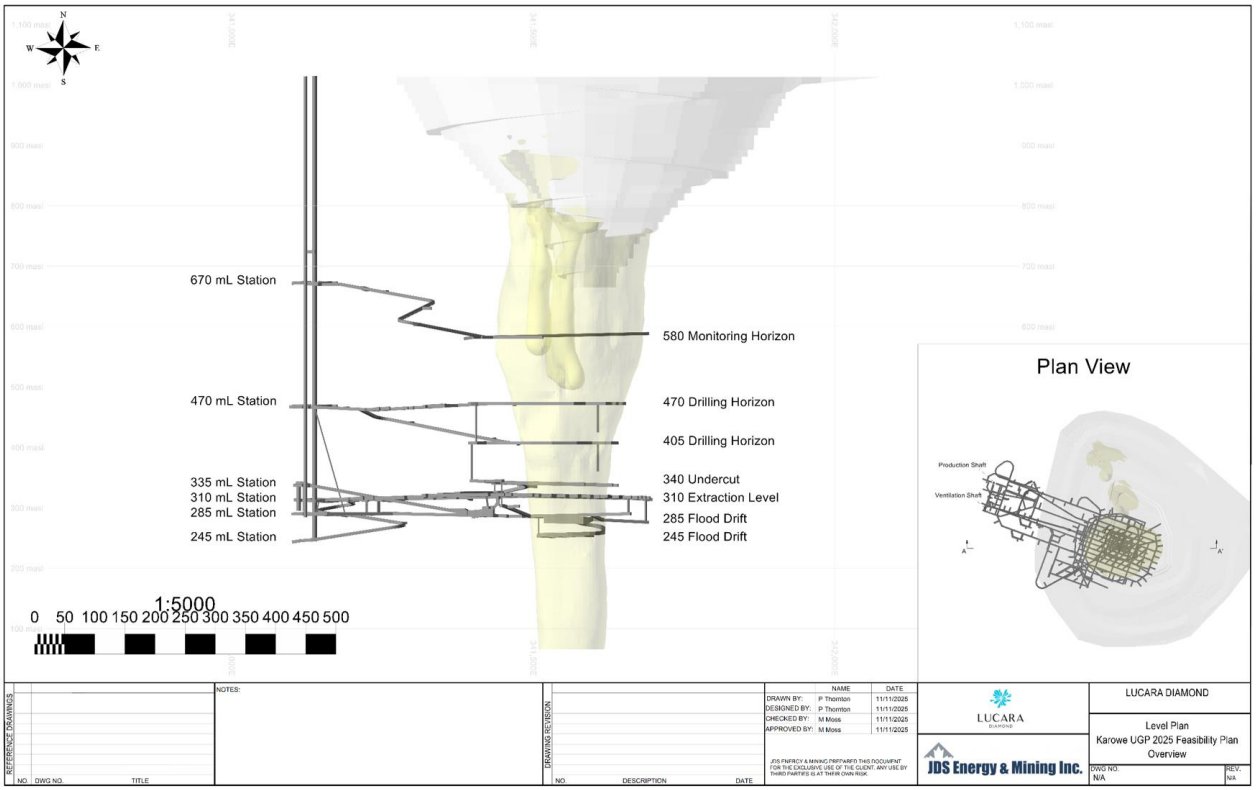
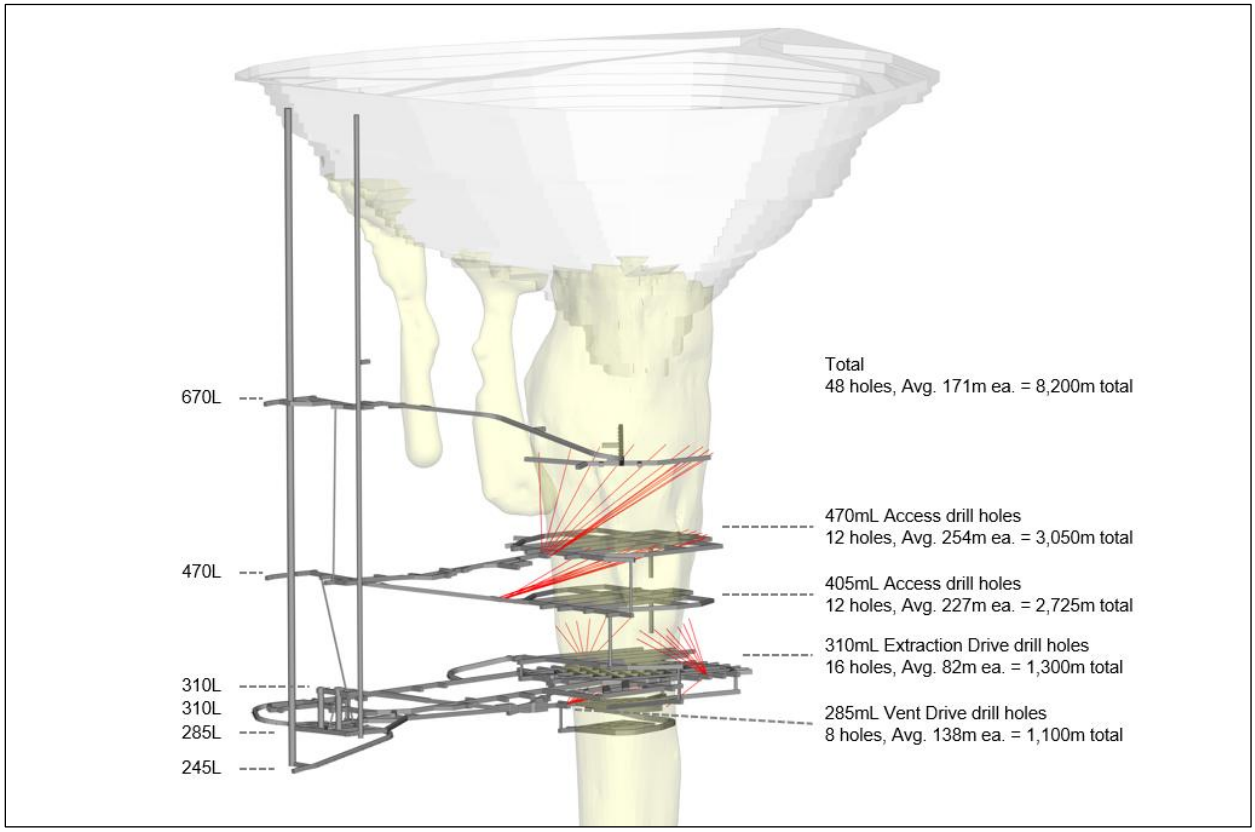


Figure 16-34: UG Drainage Gallery in 2023 Mine Design



16.4.3 Hydrogeologic Data Review, Gathering, and Analysis

No hydrogeologic investigations were conducted to gather additional hydrogeologic parameters, such as hydraulic conductivity (*K*) values, since 2024. The *K* values along the depth from past packer testing were presented in the 2023 FS report (Doerksen et al., 2024).

The geologic setting, as shown in Figure 16-31 and Figure 16-32, is typical in the region and similar to those at the Orapa and Letlhakane OPs. 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 metre per day (m/day), which is considered to be low-permeability porous media. The OP operation has been mined through the Mosolotsane unit and confirms that the sandstone units are low-permeability groundwater units. The mudstone coal formations (MUDTLH18 and Coal in Figure 16-31 and Figure 16-32) lie between the sandstone unit and granite unit. The majority of the measured *K* values in the mudstone and coal units are below 0.01 m/day, which can be considered as very low- to low-permeability geologic units. The measured *K* values of the Mea and granite units mostly range from 0.01 to 0.1 m/day, and they are considered low-permeability geologic units; however, the Mea and weathered granite are

quite variable in terms of water flows, and permeability and/or fracture water inflows could be considerably higher in certain discrete areas.

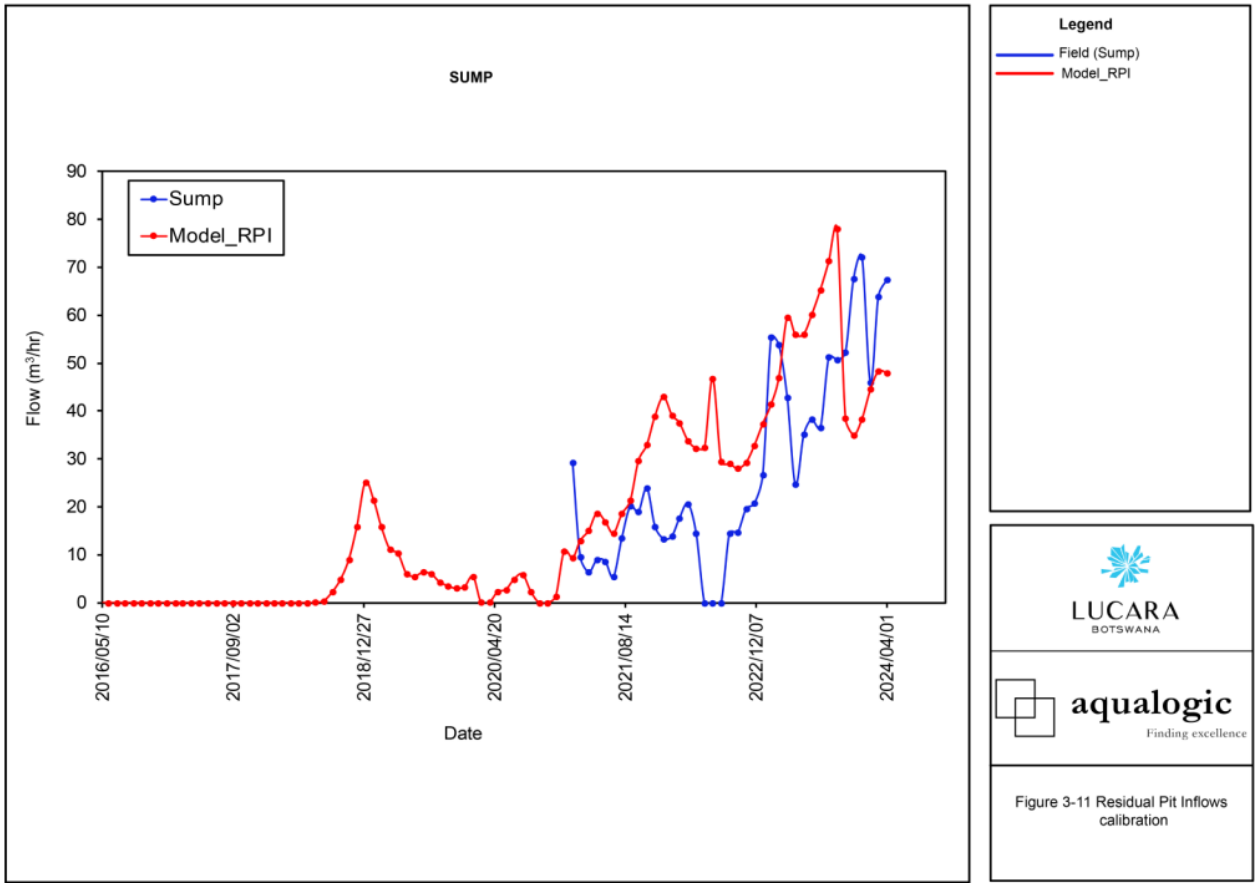
The 2019 FS report (Doerksen et al., 2019) assumed a large permeable horizon between the Mosolotsane formation and the mudstone unit based one abnormally high measured *K* value. This formation was later safely crossed by both the shaft and open pit with little inflow and was recalibrated in the 2023 FS water model updates to correct inflow predictions. Similar to the 2023 FS, this permeable horizon was not included in the 2025 FS Model.

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. As shown in Figure 16-35, the measured pumping rate from the sump ranges from 0 to 70 cubic meters per hour (m^3/hour). It appears that the sump rate shows an increasing trend starting 2023, which could be attributable to pit deepening. Also shown in the figure is the simulated seepage rate or residual passive inflow (RPI) from the 2025 Flow Model. It should be noted that the simulated seepage from the 2025 Flow Model did not consider precipitation and runoff and could overpredict the seepage rate to the pit.

Figure 16-35: Pumping Rate from Pit Sump over Time



Source: Aqualogic (2025)

The increased seepage rate could be the result of saturated thickness due to the pit deepening. Figure 16-31 shows the simulated phreatic surface along the east-west section as of October 2025. The simulated phreatic surface is about 32 m above the pit bottom, which could be the main reason for the increase in the seepage rate.

Figure 16-32 shows the simulated phreatic surface along the east-west section of the pit by the end of the Cut 2 pit in 2026. Based on the existing and planned dewatering program, the predicted phreatic surface is about 44 m above the pit bottom. Consequently, it is likely that the seepage rate could be higher than the currently observed value. However, because the future pit excavation is in the low-permeability mudstone formation, the increase of seepage to the future pit is expected to be relatively small.

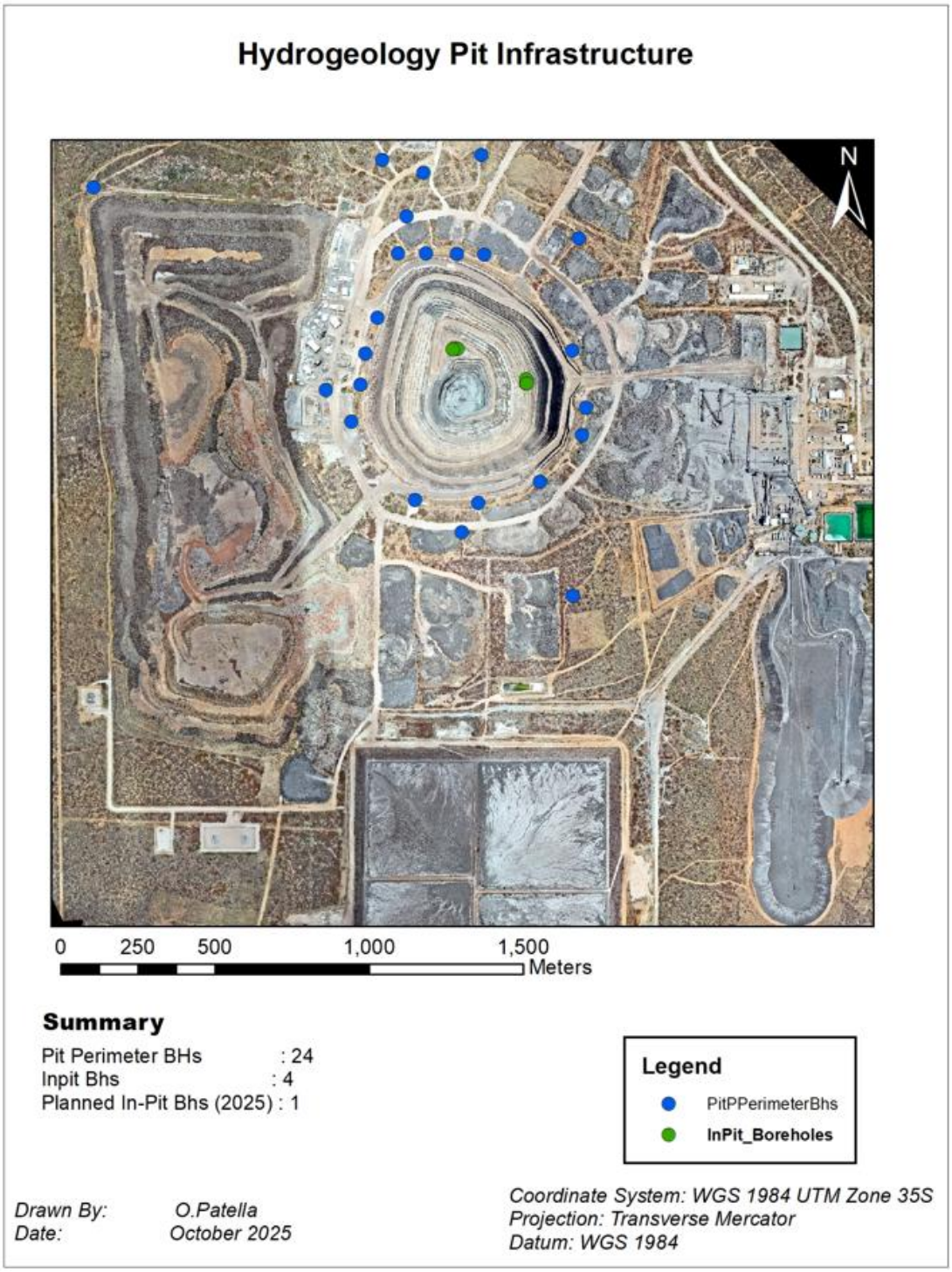
16.4.4.2 Dewatering Boreholes

Figure 16-36 shows the active dewatering boreholes for the OP operation at the Karowe Mine. As of 2025, there are a total of 28 dewatering boreholes, with 24 pit perimeter dewatering boreholes and 4 in-pit dewatering boreholes. Karowe has tried to install in-pit dewatering boreholes since 2020. However, 8 in-pit dewatering boreholes were inactive as of 2024. The reasons for these inactive boreholes were due to pit excavation, such as being mined out and losing access.

The total dewatering rates over time are shown in Figure 16-37. The total dewatering rates are as high as 390 m^3/hr in 2017. Since 2017, the dewatering rate shows a decreasing trend and stabilizes around 200 m^3/hr in 2024 (Aqualogic 2025).

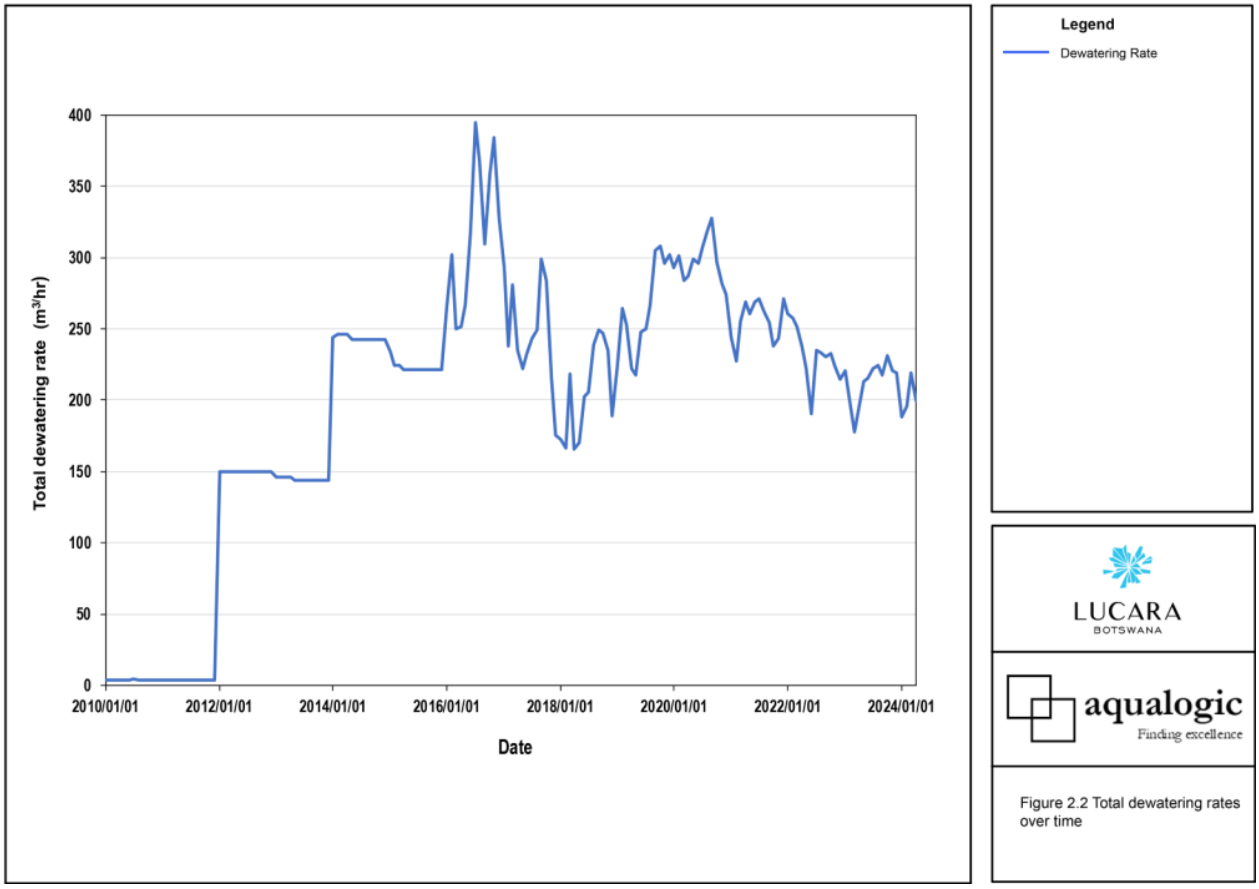
All these boreholes are for the dewatering and depressurization of the OP operation.

Figure 16-36: Active Dewatering Boreholes



Source: Lucara (2025)

Figure 16-37: Total Dewatering Rates over Time



Source: Aqualogic (2025)

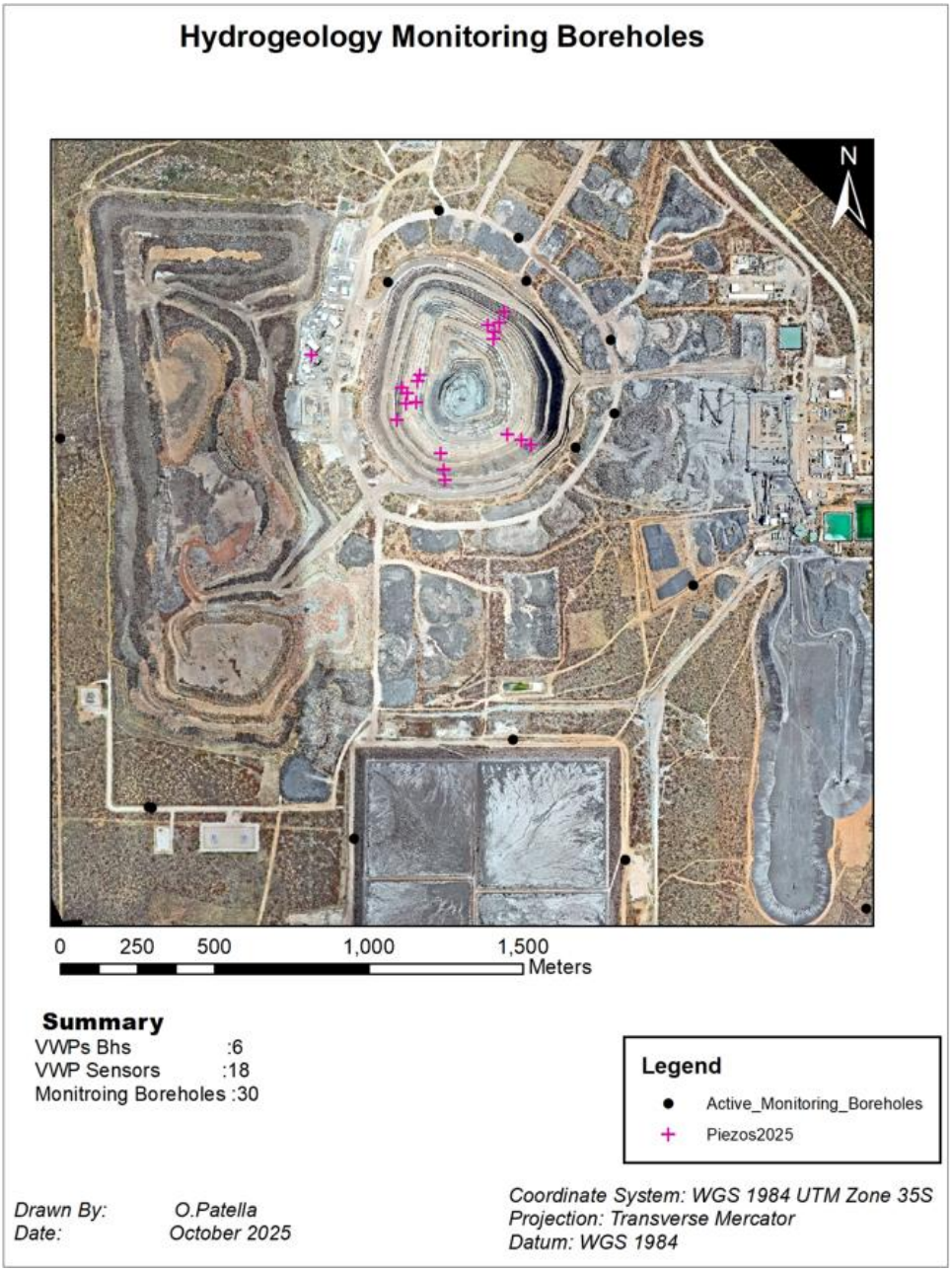
16.4.5 Groundwater Monitoring and Groundwater Levels

The groundwater levels have been monitored through monitoring boreholes and grouted vibrating wire piezometers, as shown in Figure 16-38. Almost all monitoring boreholes are located in sandstone units, with a limited number of monitoring points in mudstone and only one monitoring point in the Mea unit. Based on Aqualogic (2025), the measured water levels in the Mosolotsane and mudstone formations are around 800 masl. The Ntane sandstone shows higher water levels than the underlying Mosolotsane and mudstone formations. Detailed discussion and summary of water levels can be found in Aqualogic (2025).

Artesian flow has been observed during investigative drilling near the shafts and from within the OP, which intersected the Mea and terminated in the granite unit. These past field observations indicate that groundwater heads in the Mea and granite could be higher than the ground surface and the pit slope.

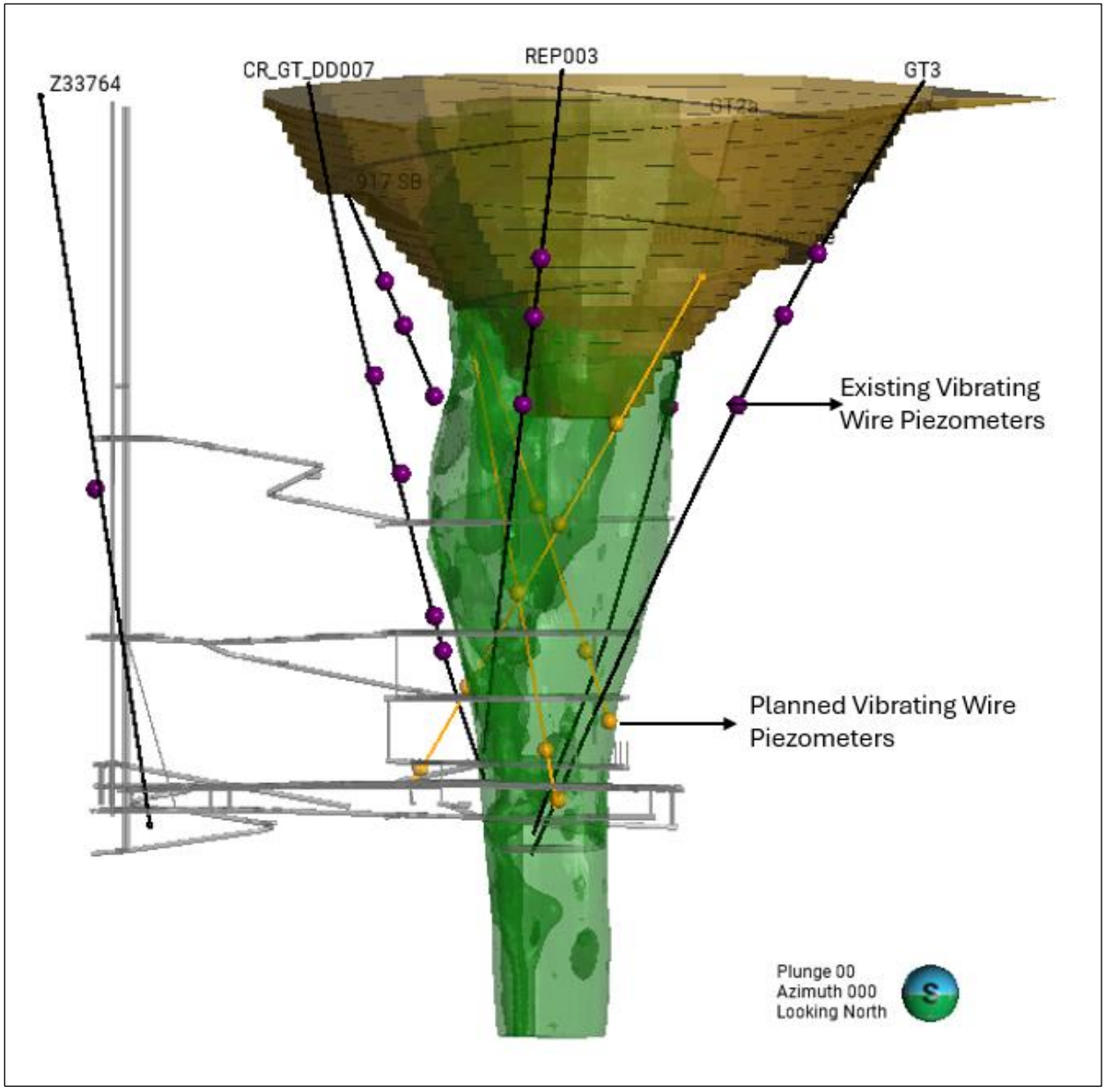
To address limited water level measurements in the granite formation an additional planned nine (9) vibrating wire piezometers (Table 16-13) will be installed between 2026 and 2027 near the kimberlite / host rock contacts, as illustrated in Figure 16-39.

Figure 16-38: Existing Monitoring Boreholes



Source: Lucara (2025)

Figure 16-39: Existing and Planned Groundwater Monitoring Instrumentation



Source: Lucara (2025)

Table 16-13: Planned Locations of Monitoring Points

BH ID	Sensor (mL)	Sensor ID	Lithology Monitored
2025-010-GGT-KDM	306.4	LG_GGT010	Leached Granite
	355.1	Kim/GR_GGT010	Kim/ Granite Contact
	512.4	KIM_GGT010	Kimberlite
2025-004-GGT-KDM	384.3	LG_GGT004	Leached Granite
	453.9	Kim-GGT004	Kimberlite
	599.6	Kim01-GGT004	Kimberlite
2025-027-GGT-KDM	335.6	GRN-GGT027	Granite
	417.8	LGRN_GGT027	Leached Granite
	681.0	KIM/TLB_GGT027	Kim/ Tlapana Contact

Source: Lucara (2025)

16.4.6 Hydrogeochemistry and Mine Water Quality

In the 2019 FS, the natural baseline water quality from the regional Stormberg Basalt-Ntane contact water strike had a total dissolved solids (TDS) signature of 1,500 mg/L to 2,000 mg/L. The deep granites had saline water with 25,000 to 33,000 mg/L TDS. The measured TDS from the recent shaft sinking is as high as 46,000 mg/L, which further confirms that the groundwater in the granite and Mea formation can be categorised as brine water type. Water quality results from ongoing monitoring are summarised 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 was last updated by Aqualogic in 2025 (2025 Flow Model). The 2025 Flow Model did not update the geology layers. Geological structures were updated based on the 2024 SRK structural model, mainly mapping new features inside shafts that act as groundwater conduits

The 2025 Flow Model was reasonably calibrated to both measured water levels and seepage rates to the open pit and UG workings. Detailed description of the 2025 Flow Model is presented in Aqualogic (2025).

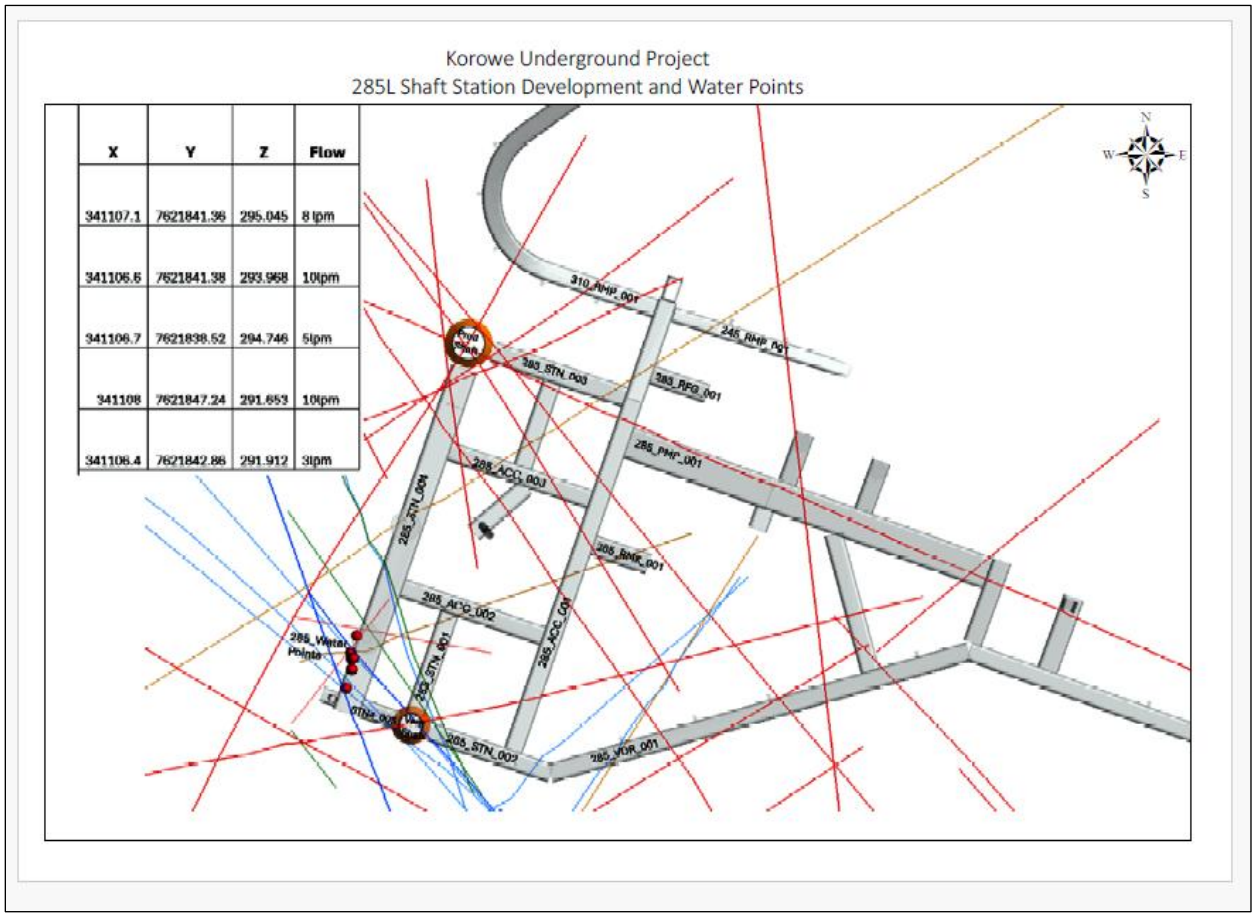
16.4.7.2 Observed Inflow to Mine Workings

The groundwater inflow encountered in the Mea and granite formations is mainly controlled by fractures under the high groundwater head. Previous encounters with water strikes or un-grouted exploration holes have provided large initial inflow rates that decrease to a low, but steady flow

rate. This pattern of observation indicates that the large inflow was mainly from the localized groundwater storage, and continuous regional groundwater recharge is limited.

Figure 16-40 shows the measured flow rate during the development of the skip loading station, located at 285 masl in the granite formation. The measured flow rate ranges from 3 lpm to 10 lpm, or from 0.2 to 0.6 m^3/hr. Based on communication with JDS and Lucara, these observed flow rates were typical during the development of the access drifts. Currently, the encountered inflow during the UG development is planned to be grouted until 2027, when the pumping system is completed to handle spikes in groundwater inflow.

Figure 16-40: Observed Flow During Drift and Access Development



Source: Lucara (2025)

16.4.7.3 Pore Pressure Target for Slope Stability Requirement

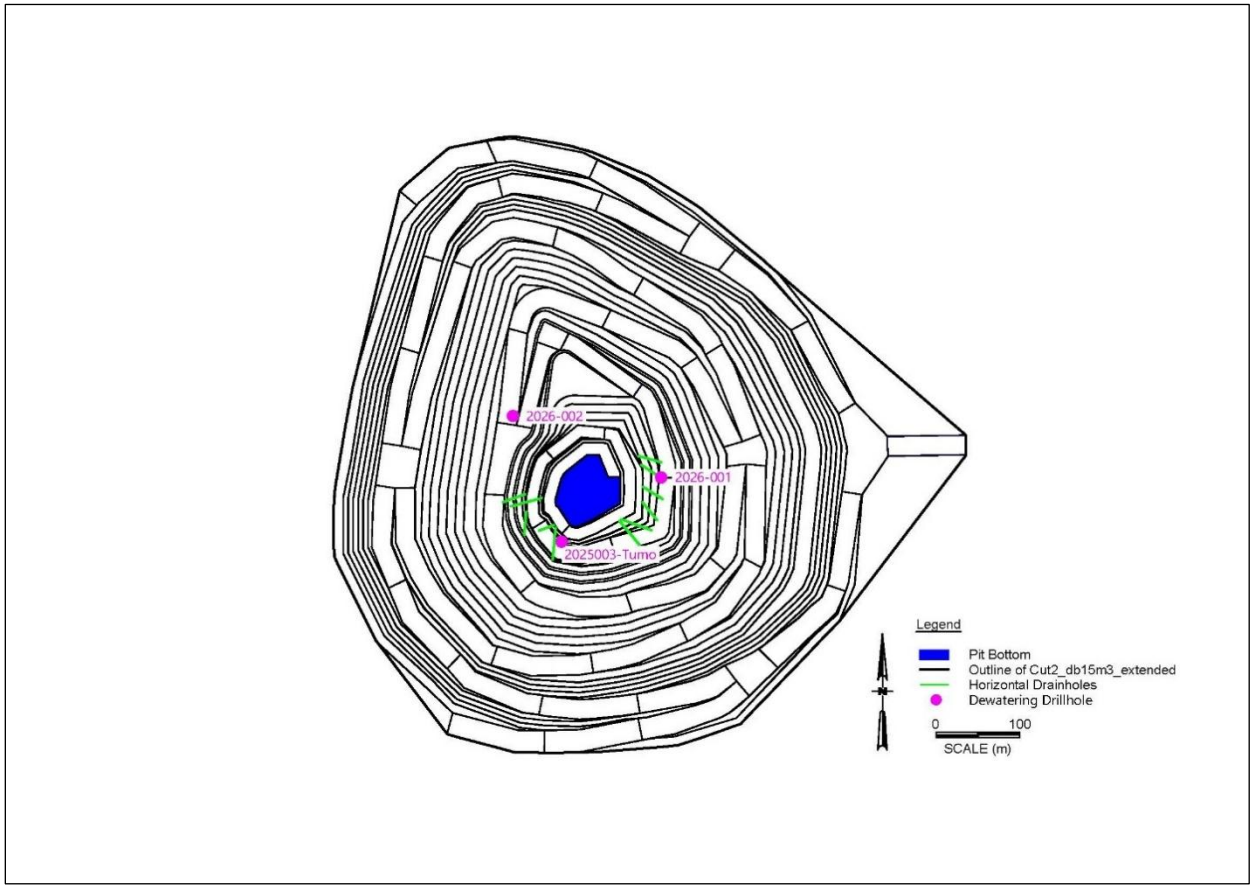
The QP conducted a stability evaluation of the Karowe Cut 2 pit bottom steepening in June 2025 (ITASCA 2025). The following are the main findings and recommendations:

1. Predictive modelling of the Cut 2 pit bottom steepening identified a marginally stable wedge (factor of safety [FoS] ≈ 1.05) on the west wall, bounded by the kimberlite contact (SRK_Fault_NS02) and a flow contact (SRK_Kim_Discontinuity_04). This wedge, situated between ramps, is sensitive to the presence, geometry, orientation, and persistence of geological structures and pore pressure; and

2. Two key sensitivity cases were examined: phreatic surface influence and flow contact uncertainty. The analysis revealed that by locally retracting the phreatic surface by 20–30 m in the vicinity of the identified wedge, the FoS of the area increased to more than 1.3, confirming that water management is critical in this area of the pit.

In order to reduce the pore pressure to meet the FoS requirements, Karowe has planned to implement three in-pit dewatering boreholes and several horizontal drain holes in 2025 and 2026, as shown in Figure 16-41. The in-pit dewatering boreholes target the leached granite with the depth ranging from 350 to 400 m. The length of the horizontal drain holes is about 30 m.

Figure 16-41: Planned Dewatering Boreholes and Horizontal Drain Holes in 2025 and 2026



Source: Lucara (2025)

16.4.7.4 Predicted Inflow Rate to UG Mining

16.4.7.4.1 Key Parameters and Scenarios of Model Predictions

The 2025 FS Model forecasts UG operation inflow rates for the LOM, improving on the 2023 FS flow model by excluding un-grouted exploration boreholes that have since been grouted. Previously, simulated inflows through historical un-grouted boreholes drove large spikes in inflows early in the mine life. The 2023 FS flow model was used to identify and grout high-risk sites between 2024 and 2025. The 2025 FS Model omits these grouted boreholes from simulations. KDM should regularly review and grout open drillholes to prevent increased groundwater inflows to manage the significant risks of unexpected inflow.

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

- Dewatering Boreholes:
  - All existing dewatering boreholes and recommended dewatering boreholes as presented in Figure 16-36 were incorporated. 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 planned future dewatering boreholes, the initial dewatering rate was assigned to be 10 m^3/hr.
- OP:
  - The 2025 updated OP mine plan was incorporated.
- UG Mining:
  - The 2025 updated UG mine plan and drainage gallery as illustrated in Figure 16-34 were incorporated. Both shafts have been either fully grouted or concrete lined and were treated as impermeable in the model; and
  - The predicted ground displacement due to the caving operation was incorporated in the model.
- Model Scenarios:
  - Two model scenarios were simulated. Scenario 1 assumed that the underground mine workings will be grouted until August 2027. Because there is only 50 m^3/hr pumping capacity until August 2027, grouting could be required to manage the groundwater inflow during the development of the underground mine workings; and
  - Scenario 2 assumed that there is no grouting except for the grouting of both the V/S and P/S. This scenario was simulated to assess if the inflow to the mine workings exceeds 50 m^3/hr before August 2027.

16.4.7.4.2 Predicted Inflow Rates

Figure 16-42 and Figure 16-43 show the predicted total inflow rate and inflow rates to different underflow components to the mine workings for Scenarios 1 and 2, respectively. The following observations can be made:

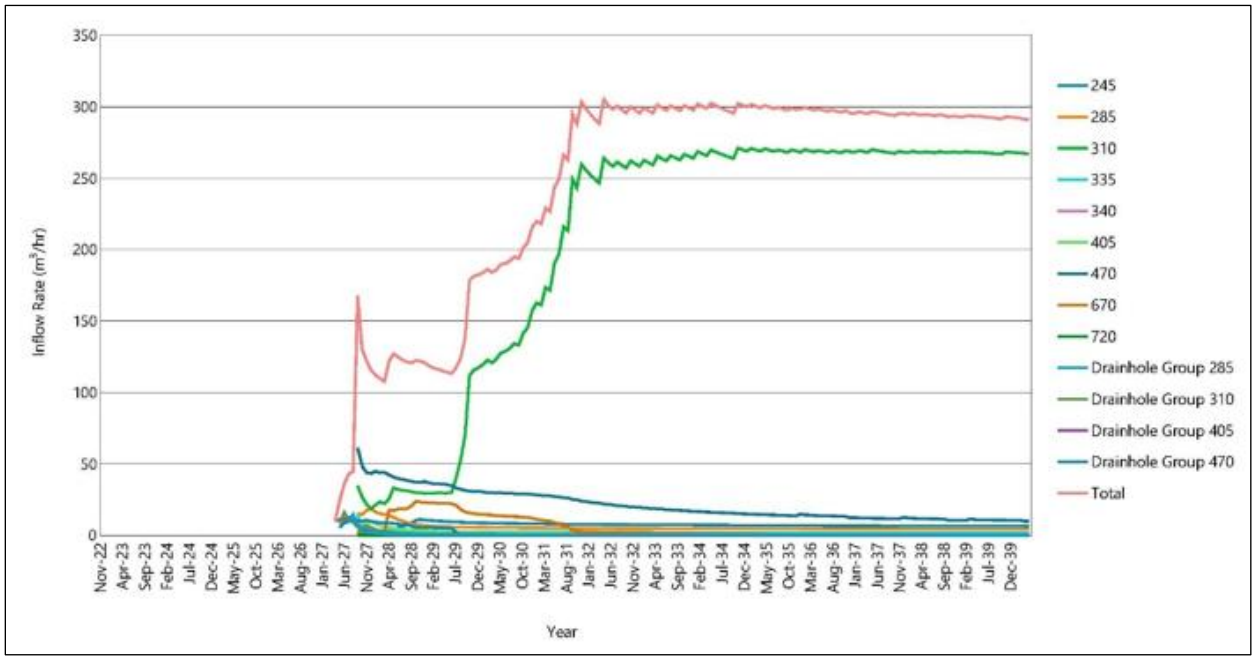
1. The simulated inflow rate under the un-grouted scenario in August 2025 is around 52 m^3/hr, which is comparable to the measured inflow rate of 33 m^3/hr that was provided by Karowe mine in 2025. It should be noted that the measured inflow rate is instantaneous, as the intercepted seepage was subsequently grouted;
2. The maximum total inflow rate over the LOM is around 300 m^3/hr for both Scenarios 1 and 2;

3. Under the no-grouting scenario, the simulated total inflow rate could be as high as 100 m^3/hr by August 2027, which exceeds the current pumping capacity of 50 m^3/hr. This result suggests that grouting is necessary until sufficient pumping capacity is available; and

4. The maximum inflow rates to Drainhole Groups 470, 405, 310, and 285 are 10.5, 12.9, 15.9, and 14.3 m^3/hr, respectively. The higher inflow in the higher drain groups is a result of higher pressure in the rock mass in the upper drain galleries and lower pressure in the rock mass in the lower drain galleries.

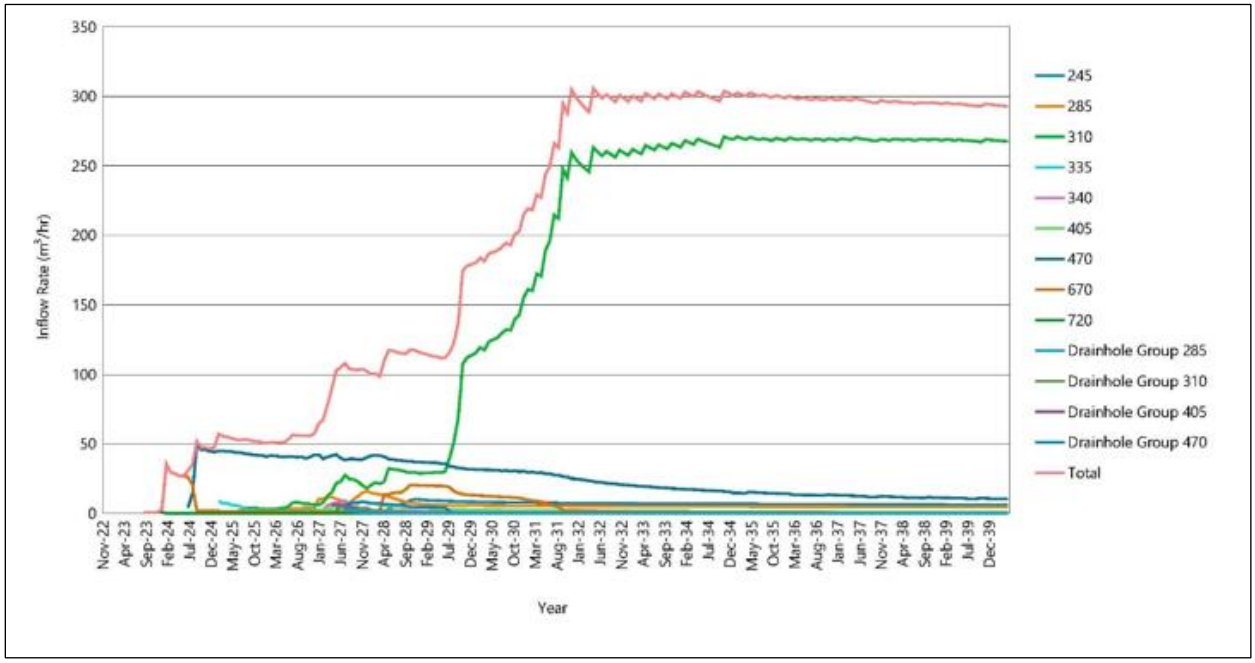
The maximum inflow rates to Mine Levels 245, 285, 310, 335, 340, 405, 470, 670, and 720 are 11.1, 18.5, 271.1, 14.0, 2.7, 9.4, 61.1, 23.6, and 0.0 m^3/hr. respectively. The high inflow rate to Mine Level 310 corresponds to the decrease of inflow rate to Drainhole Group 470 when the cave zone intercepts the stope mining zone.

Figure 16-42: Simulated Inflow Components with Grouting until August 2027



Source: Itasca (2025)

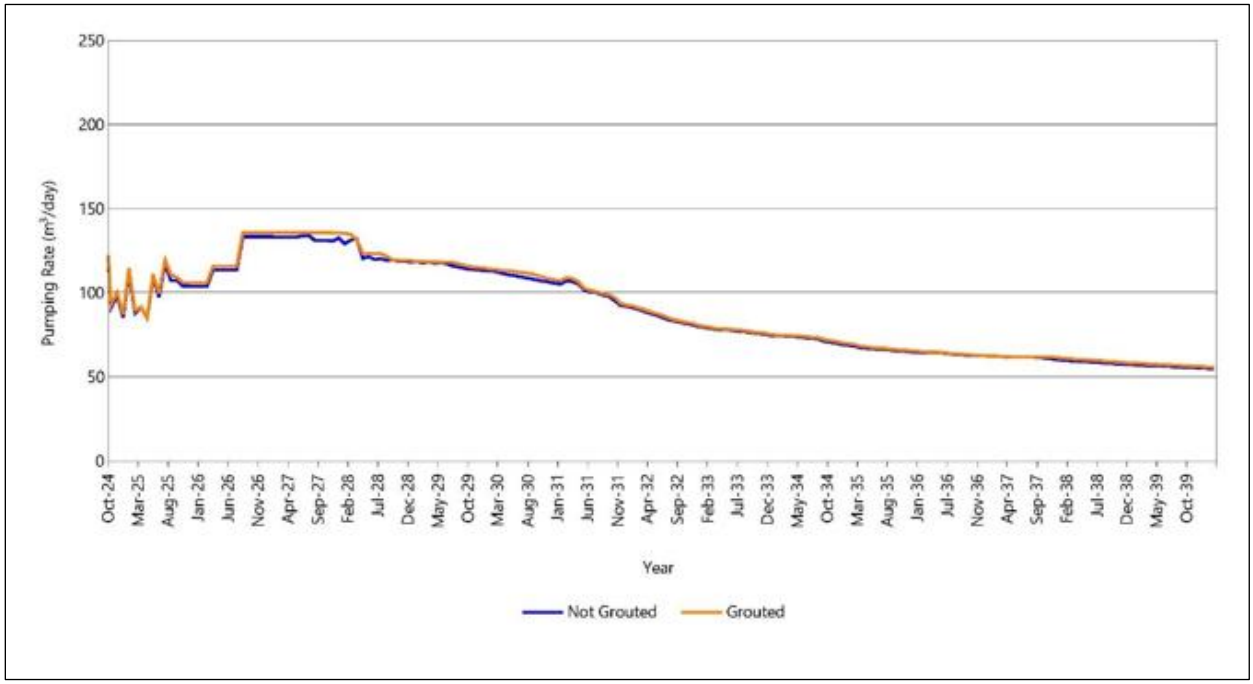
Figure 16-43: Simulated Inflow Components without Grouting



Source: Itasca (2025)

Figure 16-44 shows the predicted dewatering rate for the dewatering boreholes over the LOM for both Scenarios 1 and 2. The dewatering rate from the dewatering system increases to 130 m^3/hr in 2026 as the result of commissioning of the planned dewatering boreholes. After 2087, the dewatering rate decreases over the UG mining operation for both scenarios to about 50 m^3/hr. Figure 16-44 suggests that the UG mining will reduce dewatering rates of the active dewatering boreholes of the OP. Because of the need to maintain the dry open pit condition and to depressurize the pit slope, it is critical to maintain surface dewatering operations to reduce the seepage to the OP and meet the pore pressure requirement for slope stability during the life of the UG operation. The predicted dewatering rate is not sensitive to the short-term grouting.

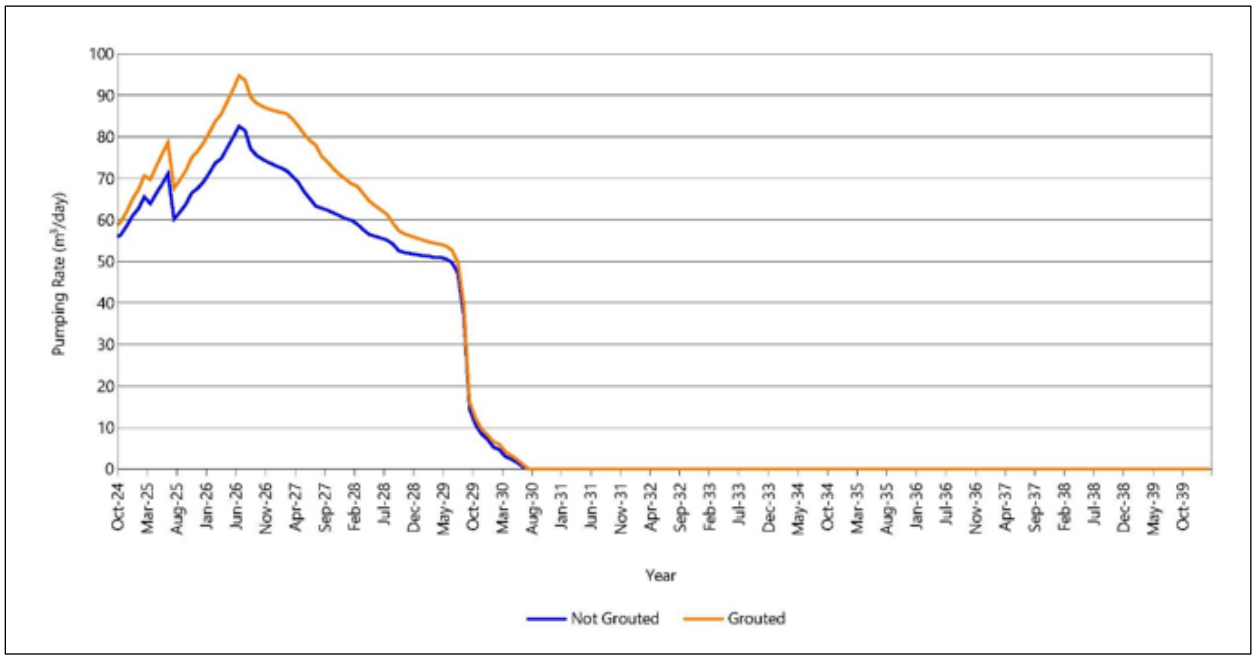
Figure 16-44: Simulated Dewatering Rate from Dewatering Boreholes over LOM for Scenarios 1 and 2



Source: Itasca (2025)

Figure 16-45 shows the predicted seepage to the pit over the LOM for both Scenarios 1 and 2. The seepage rate decreases to zero after 2030 when the stope mining intercepts the cave zone. The seepage rate from the grouting scenario is slightly higher than the non-grouting scenario because of the depressurization effect of non-grouting.

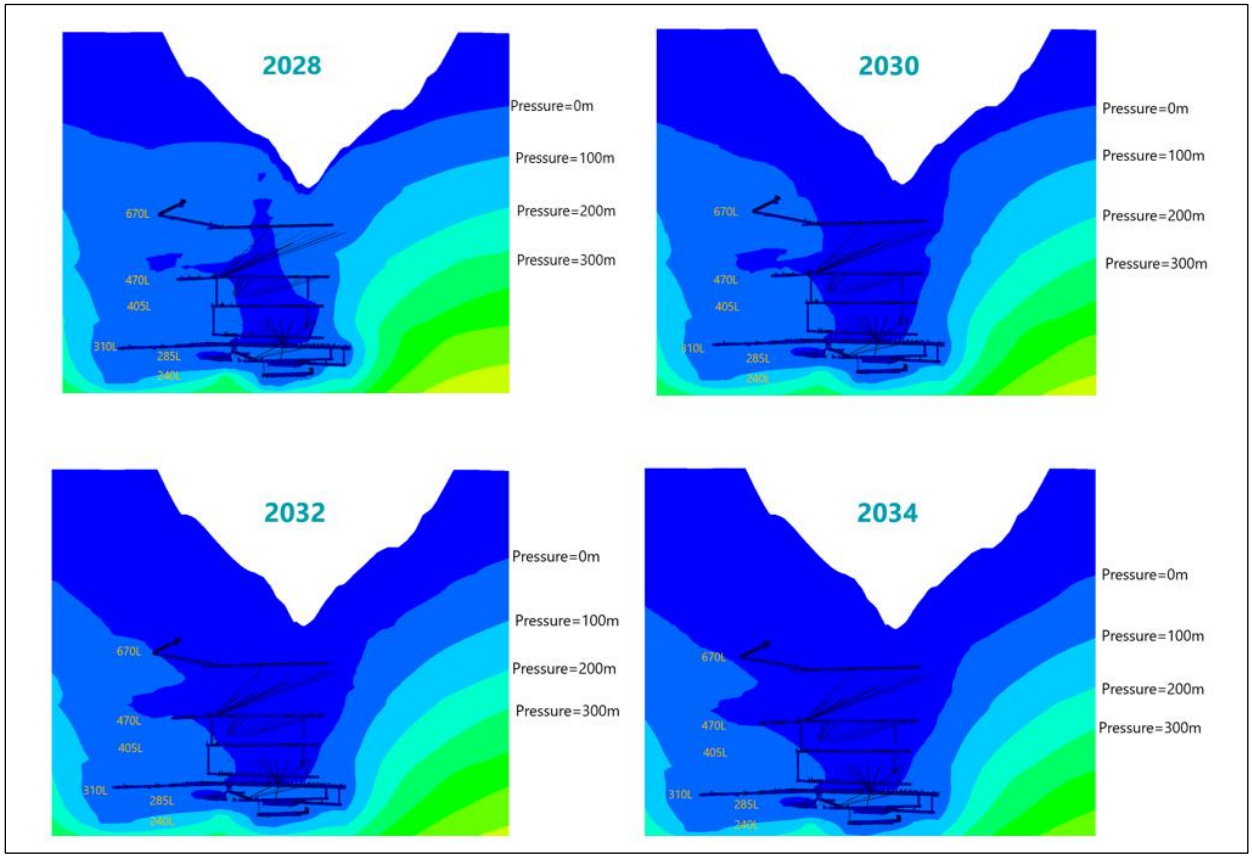
Figure 16-45: Simulated Seepage Rate over LOM for Scenarios 1 and 2



Source: Itasca (2025)

Figure 16-46 shows the pore pressure distributions for the country rock and kimberlite in 2028, 2030, 2032, and 2034 under Scenario 1. Also shown in the figure are the main levels of the mine workings and drainage gallery. The pore pressure distribution over the LOM shows Level 310 is under the saturated condition with the pore pressure around 100 m of water column, or 1 megapascal. The saturated condition at this level leads to the higher inflow to the 310 Level in comparison to other mine levels.

Figure 16-46: Simulated Pore Pressure Distributions over LOM for Scenario 1



Source: Itasca (2025)

16.4.8 Assumptions and Uncertainties

Though the assumptions have been discussed in various sections, the key assumptions and uncertainties related to the predicted pore pressure and 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 18.2

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 granite units. The lack of hydrogeologic data poses uncertainty to the groundwater

management for the UG mining operation and leads to uncertainty of the groundwater flow model predictions.

Recent field geologic investigations indicate that leached granite is present in the vicinity of the kimberlite. Karowe Mine is currently updating the geologic model to define the extent of the leached granite. The extent of the leached granite could affect the inflow rate to the mine workings because the leached granite is considered to be more permeable than the rest of the granite formation. Based on the observed flow rate from the underground development and from the drilling from the open pit, it appears that flow from the leached granite is limited because the source of groundwater flow to the leached granite is from the regional granite formation. However, if the fracture corridor is permeable and present regionally, the interception of the leached granite and fracture corridors could lead to a higher inflow risk. Assessment on the potential interception of the leached granite and fracture corridors should be conducted, and the potential high flow risk zones should be identified. Hydraulic testing, such as packer testing or pumping tests, should be planned, and groundwater level monitoring points in these risk areas should be implemented.

16.5 Mine Design Criteria

16.5.1 Basis

Mine design, schedules, and plans incorporate the current state of mine construction as at end of September 2025.

16.5.2 Production Rates and Schedules

Mine schedules are structured around continuous operations, utilizing two 12-hour shifts per day, with an effective working time of 15-18 hours daily (task dependent). Shaft schedules reflect an available hoisting window of 18 hours per day. Mobile equipment is expected to achieve a mechanical availability rate of 80%. Production targets for mill feed rates are set at 2.85 million tonnes per annum (Mtpa), with design capacities accommodating up to 10,000 tonnes per day (t/d) to address both mechanical and operational availabilities. The mine is deemed to have reached commercial production status once it sustains 75% of the targeted production level over a consecutive 30-day period.

16.5.3 Material Properties

The Mine design will rely on physical properties stored within the resource block model, including volumes, densities, grades, rock mass ratings, and mined status. Where information cannot be read directly from the resource block model the values listed in Table 16-14 are assumed.

Run-of-Mine material will be crushed, conveyed, and hoisted from underground to surface. Table 16-14 lists the material properties of Kimberlite to be handled by the circuit.

Table 16-14: Material Properties

Key Operating Parameters	Unit	Value
Kimberlite Ore Density	t/m^3	2.9
Kimberlite Ore Moisture	%	8.0
Kimberlite Swell Factor	%	36.4
Crushed Ore Bulk Density	t/m^3	2.10
Angle of Repose	deg	38
Crusher Work Index, CWi	kWh/t	16.00
Bond Ball Mill Work Index, BWi Weighted 75%	kWh/t	24.00
Abrasion Index	g	0.26

16.5.4 Mine Development

Lateral development will be completed using trackless machinery and drill and blast methods. Vertical development will be completed using a combination of blind sink, raisebore, long hole drill and blast, and slipping methods determined by the length, diameter, and function of the excavation.

Drift profiles are defined by the size of equipment used to develop the mine, as well as the services required in the drift. Local Botswana regulations do not specify drift profile dimensions or safe distances. For planning purposes the Karowe UGP will leverage British Columbia, Canada Mine Regulations.

Drift width is designed 2.0 m over the widest machine working in the area, rounded up to the nearest 0.5 m, to account for 1.0 m clearance on either side of the machine.

Drift height is designed 1.9 m over the tallest machine working in the area, rounded up to the nearest 0.5 m, to account for vent ducting, ballast, and clearance below services.

Refer to Table 16-15 for the primary development equipment list, with assumed Equipment Manufacturer for design purposes. Based on this fleet, the 21 t production loader and 2-Boom Jumbo drill determine minimum drift dimensions of 5.5 m W x 5.5 m H at the extraction level. On levels where the production loader is not used 5.0 m W x 5.0 m H drives will be applied.

Table 16-15: Equipment List

Equipment	Manufacturer/Model	Machine Width	Machine Height	Min. Drift Width	Min. Drift Height
LHD (17 t / 7.0 m^3)	Sandvik - LH517	3.1	2.8	5.5	5.0
LHD (21 t / 8 m^3)	Sandvik - LH621	3.2	2.9	5.5	5.0
Jumbo - 2 Boom	Sandvik - DD422i	2.3	3.2	4.5	5.5
Long-hole Drill	Sandvik - DU311-TK	2.1	2.9	4.5	5.0
Secondary Breakage Drill	Sandvik - DB331	3.1	3.1	5.5	5.0
Bolter	Sandvik - DS411	2.9	3.1	5.0	5.0
Cable Bolter	Atlas Copco - Cabletec M	2.6	3.1	5.0	5.0

Figure 16-47 and Figure 16-48 illustrate typical drift specifications for the Project. Refer to Section 16.7.2 for details on Shaft design.

Figure 16-47: Typical Drift Profile

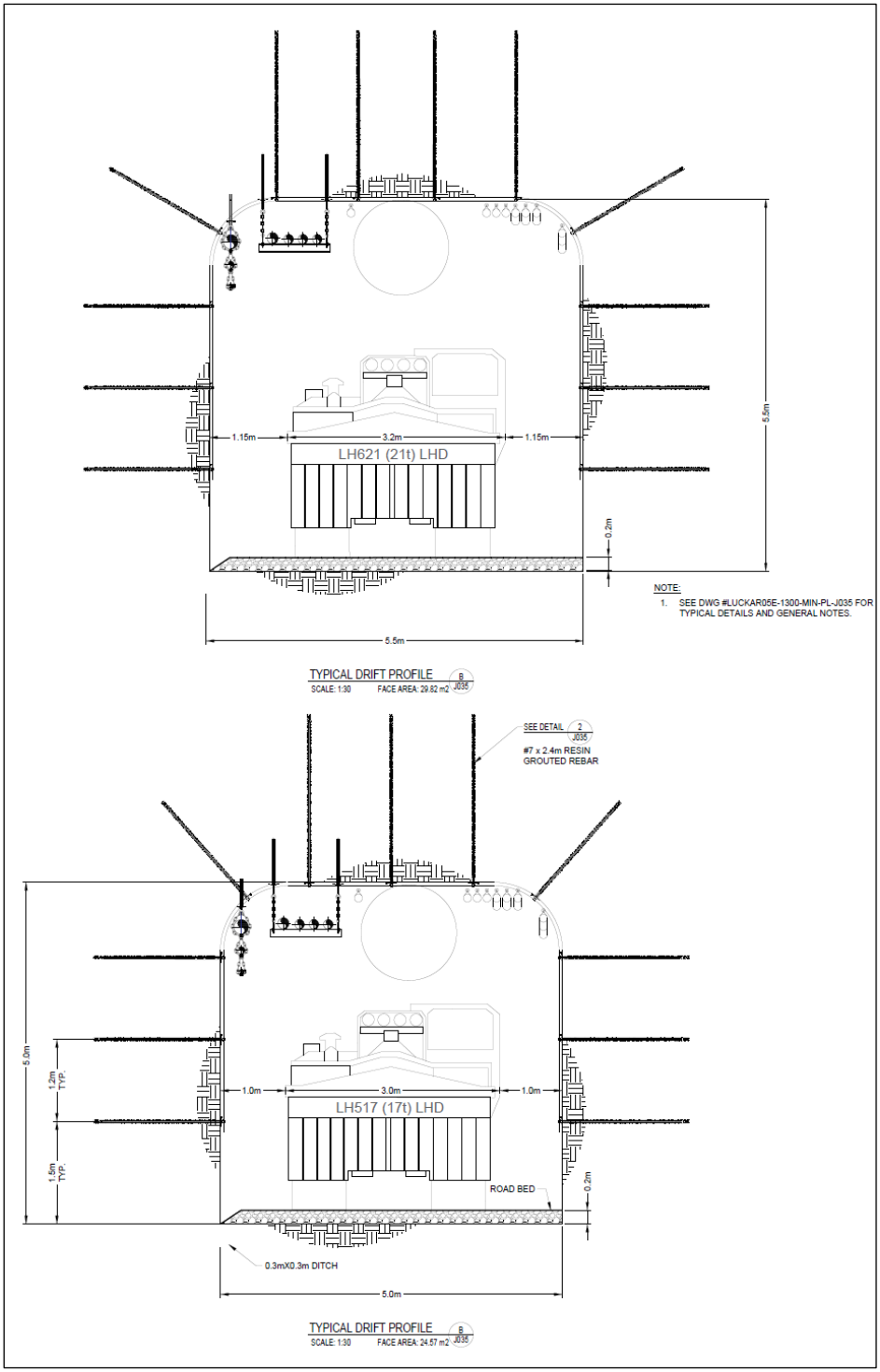
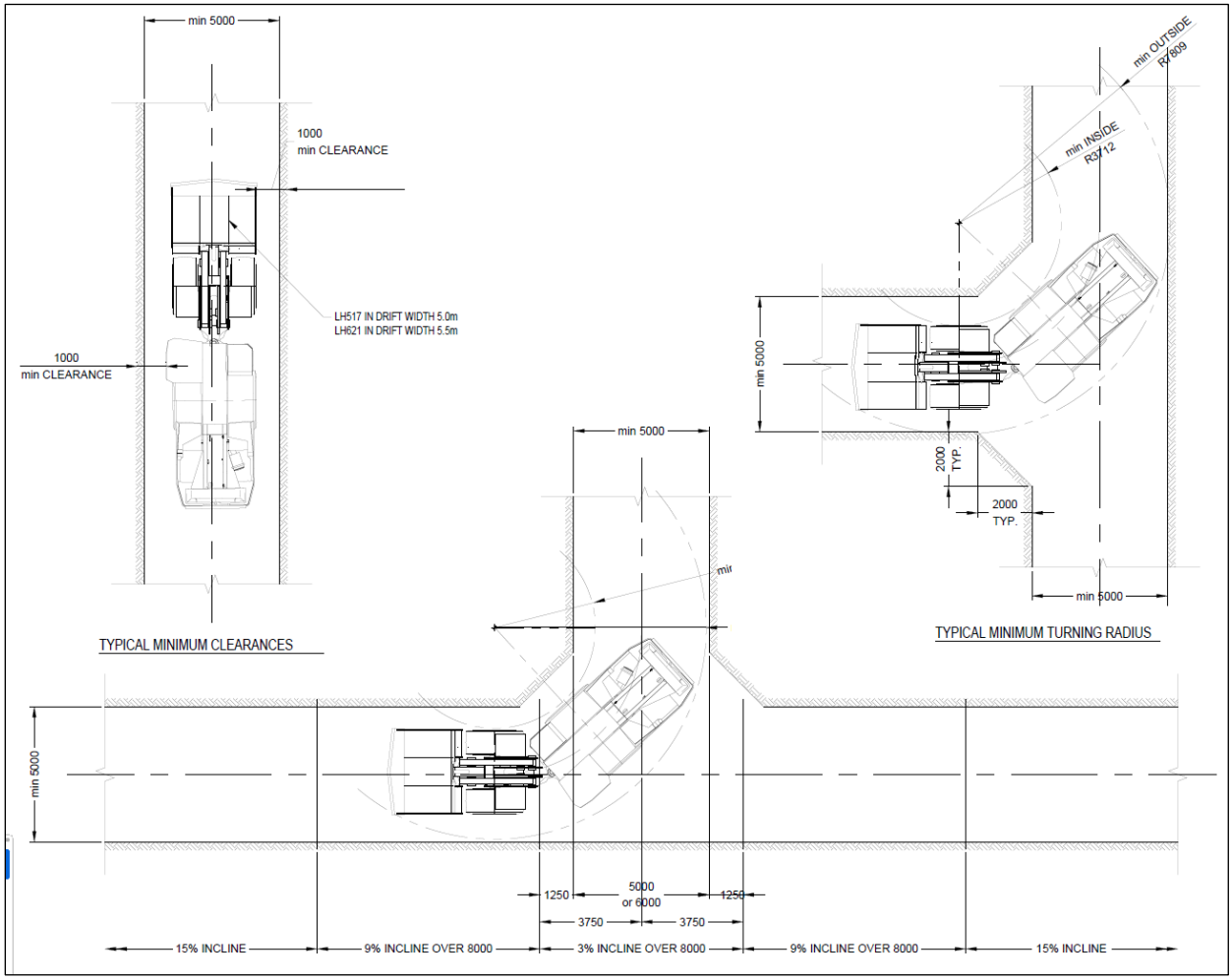


Figure 16-48: Drift Specifications



16.5.5 Mobile Mine Equipment

Mobile Mine Equipment shall be Rubber tired and Conventional Tier 3 diesel and electric / hydraulic. A mix of new and used, client owned and contractor operated equipment is currently used to construct the mine.

The Karowe UGP does not include provision for trucks. Given the relatively short (350 m) distance of the ore body to the primary tips it is estimated that trucks will not drive sufficient schedule improvements to outweigh the costs of the labour and underground infrastructure they require. As such, the mine design does not cater for truck loading, tipping, parking, or hoisting requirements.

16.5.6 Contractor Support

Contractor Support shall be utilised for:

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

Owner's Team shall be utilised for:

- Drawbell Operations;
- Comminution and Material Handling; and
- Shaft operation and maintenance (following construction completion).

16.6 Mining Methods

16.6.1 Open Pit

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. The mining contract has a mixed fleet of trucking, hauling, drilling, support, and ancillary equipment available on-site.

OP mine operations are expected to terminate in April 2026 at an elevation of 665 masl (Cut2_db15 m^3_extended, Karowe 2025). The mine currently has over two years of stockpiled reserves, which will be partially 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 Underground

Underground mining was first considered in the Royal Haskoning DHV's 2017 Preliminary Economic Assessment (Oberholzer, 2017) which evaluated UG mine methods at KDM including block caving, sub level caving, and long-hole open stoping. Sub-level caving with ramp access was preferred for its economic benefits but posed geotechnical risks in weak ground. Further trade-off studies were recommended to refine UG access and mining method selection. In response, Lucara Diamonds conducted an internal study in 2018 and commissioned JDS in 2019 to prepare a Feasibility Study to reassess the optimal approach for the deposit.

The 2019 Feasibility Study evaluated several underground mining methods using data from a 2018–2019 field program to assess resource, geotechnical, and hydrogeological factors. Due to the small hydraulic radius at depth (27 m), low in situ horizontal stress, and high kimberlite strength, natural or preconditioned caving was not considered feasible, necessitating drill-and-blast assistance. This led to the development of the LHS method—a fully assisted cave involving long-hole drilling and blasting in the South Lobe, with controlled removal of material through an extraction level—which was adopted in both the 2019 and 2023 studies.

During a 2021 design evaluation, recommendations were made by the 2019 Geotechnical QP to adopt a trough-style drawpoint in exchange for the original drawbell design to improve schedule and cost. Initial geotechnical evaluations of these trough designs performed by Itasca found them to be stable and capable of meeting production requirements. As such, the trough extraction designs were introduced into the 2023 Feasibility Study Update.

Subsequent geotechnical evaluations performed in 2024 focused on confirming the orientation, spacing, size, and overall geotechnical stability of the trough design at the extraction level. Inputs to this study included updated geotechnical data, hydrological models, structural modelling, and a comprehensive drillhole database encompassing all past exploration and drilling conducted beyond the 2019 feasibility study. The results indicated that removing apex pillars, traditionally applied in a drawbell orientation, compromised the extraction ratio on the extraction drive, leading to instability of the remaining pillars between draw points and troughs. Recommendations were made to abandon the trough design and revert to the 2019 draw bell system, which employs primary and secondary apex pillars to maintain the stability of the draw points and the entire extraction level. Additionally, with the updated structural modelling, a general northwest-southeast trending structure was identified. It was recommended to abandon the offset herringbone drawpoint arrangement in exchange for a northeast-southwest facing El Teniente drawbell arrangement to cross these structures perpendicularly and optimize geotechnical stability.

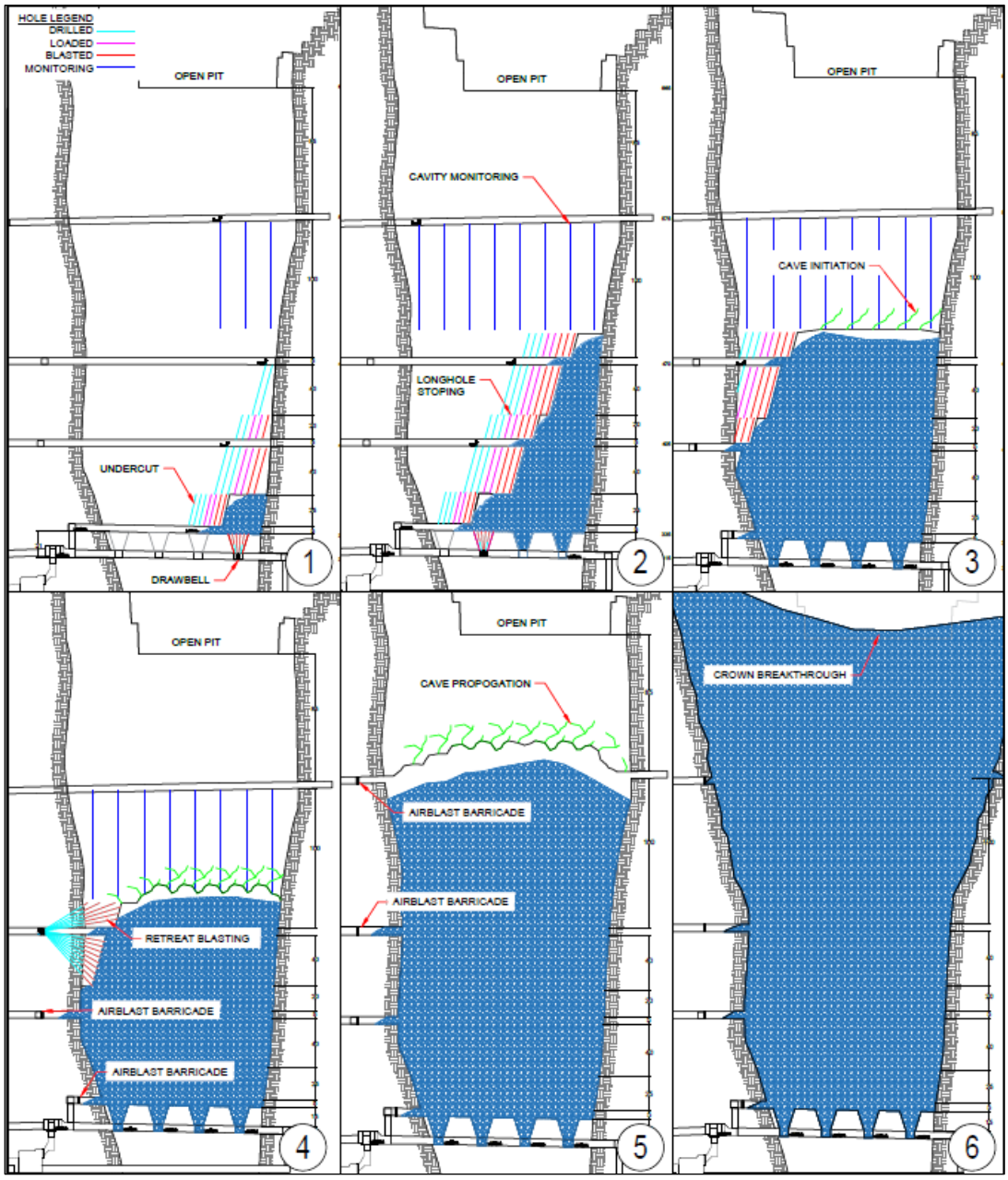
In response to recommendations made in the 2023 Feasibility Update, geotechnical predictive modelling was performed in 2024 to further examine cavability, and prepare Cave Management Plans (CMP). The result of this work confirmed previous caving evaluations made for the lower levels of the pipe – blast assistance is required - but concluded that strong potential existed for unassisted block caving from approximately 170 m above the extraction level. Hydraulically coupled cave simulations were performed to confirm the location, rate, and evolution of a natural cave within the South Lobe, further discussed under Section 16.3, and has led to the introduction of the block caving mine method following an initial stoping, or Blast Assisted Cave (BAC) campaign.

The proposed mine method is illustrated in Figure 16-49. Benefits of the proposed mining method include:

- Bottom-up mining approach schedules highest value ore first;
- Minimal development in weak, water-bearing lithologies near surface;
- Development of the UG mine is performed simultaneously with the OP operations;
- Capitalizes on low operating costs associated with block caves;

- Ease of operation after the drilling and blasting phase is complete with small UG work force; and
- Early exclusion of stormwater into the UG workings until the cave connects with the open pit.

Figure 16-49: Mine Method - Mining Sequence



16.7 Mine Design

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

- Two shafts to provide for all personnel 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 a portion of the ore body is drilled and charged;
- Instrumentation drives to allow for monitoring of the cave back;
- Interconnecting ramps and raises where required to provide personnel, 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 designs and schedules were completed in Deswik software, and in collaboration with LR4-FS5 coupled numerical model caving simulations.

16.7.1 Mine Access

The UG Mine will be primarily serviced by two shafts located 375 m North-West of the existing OP at a 110-degree bearing, and 100 m from one another. These shafts have been sunk blind using conventional drill and blast equipment and developed concurrently. The shafts are not subjected to freezing or hydrostatic lining; groundwater ingress has been effectively managed through the implementation of in-shaft grouting. Shaft sinking commenced in 2021 with final blasts taken in December 2025, and equipping scheduled for 2026. Shaft access has been selected, in part, for its ability to quickly cross lithological zones of weakness in a controlled manner. The shafts will serve as LOM access to the underground mine.

16.7.2 Shaft Design

A single P/S will provide personnel and material access underground. The P/S will be 8.5 m finished diameter, concrete lined, and equipped with rock hoisting skips, personnel 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.

A single V/S of 6.0 m finished diameter, concrete lined, shall serve as the primary return air path and secondary egress (refer to Section 16.8.14). No services, conveyances, or ladderways will be equipped in the V/S.

16.7.2.1 Shaft Siting

Shaft locations were selected based on the following criteria.

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: Strain analysis along production and ventilation shafts was conducted to evaluate shaft stability during LHS mining in 2020 (Itasca, 2020). Strain is a better indicator of stability than displacement, as it reflects damaging relative movements. Shaft liner stability should not exceed 0.1% strain; damage occurs at 0.07% in axial compression and 0.02–0.03% in tension. Using these criteria, modelled axial strains remained well below limits throughout the mine's life, indicating the shafts are placed safely outside excavation influence. Stress modelling confirmed significant induced stresses are localised near the pipe and do not impact the shafts. 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 to minimise fly rock induced safety risk and damage from OP operations.

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 LOM to service all personnel, material, and rock movement between surface and UG.

Headframe and collar construction was completed in 2022 as shown in Figure 16-50. P/S and V/S headgears are 61 m and 41 m tall respectively.

Figure 16-50: Production (left) and Ventilation (right) Shaft Headgear



Source: JDS (2024)

16.7.2.3 Shaft Capacity

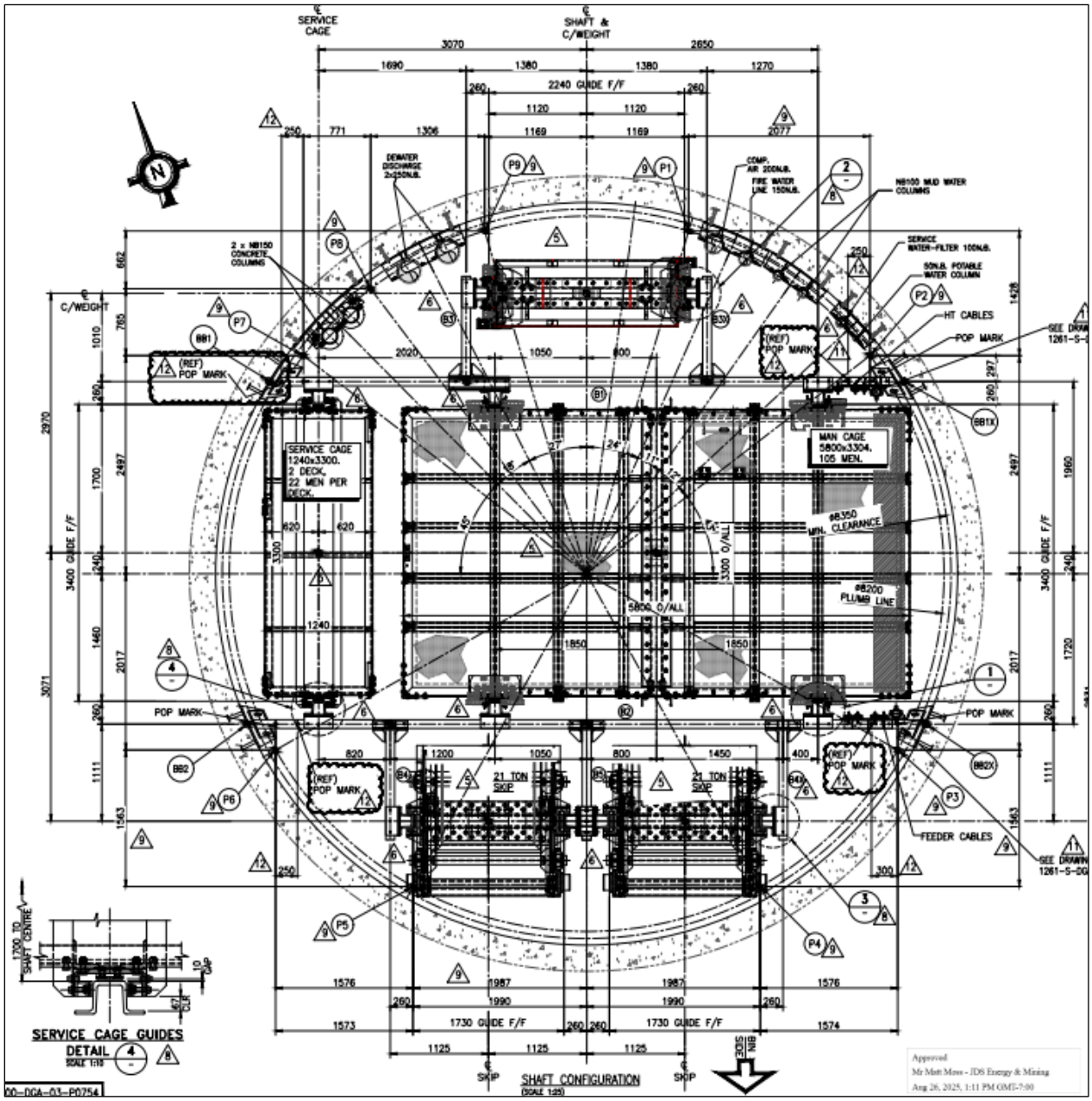
The P/S shaft has the following design elements and capacities:

- 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 people or 16.5 t in-cage, and 26 t with a heavy lift bridal; and
- One (1) auxiliary cage with 16- person capacity.

The P/S skips, cages, and counterweight will all operate on fixed steel guides. Shaft internal arrangement can be seen in Figure 16-51 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 16-51: Production Shaft Internal Arrangement



Source: UMS (2025)

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 16-16. 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 16-16: 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	Total End Load	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
Personnel and Material Cage	2	2,400	4.88	8.7	21,332	16,500	37,832	64	1,142
Aux Cage	1	700	2	6.75	4,600	1,500	6,100	26	1,003
Ventilation Shaft									
N/A									

Source: UMS (2025)

16.7.2.5 Shaft Stations

The shafts will service seven (7) shaft stations between 350 m and 750 m below surface. Several shaft stations will be connected to one another through ramps or connections while some are captive such that the only means of access to that level is through the shaft.

V/S stations are not 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 are equipped with permanent services including air, power, and water. Incoming water lines will be provisioned with pressure-reducing stations. Station protective steelwork will be used as the mounting points for thrust blocks on incoming and outgoing water lines.

Station floors are 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 are erected at the entry of each shaft station. D-plates are mounted to the walls of the station to provide lifting points for large loads and mobile equipment.

Water rings have been 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 have been installed as needed above V/S stations during the shaft sink to mitigate nuisance water.

16.7.2.6 Skip Loading

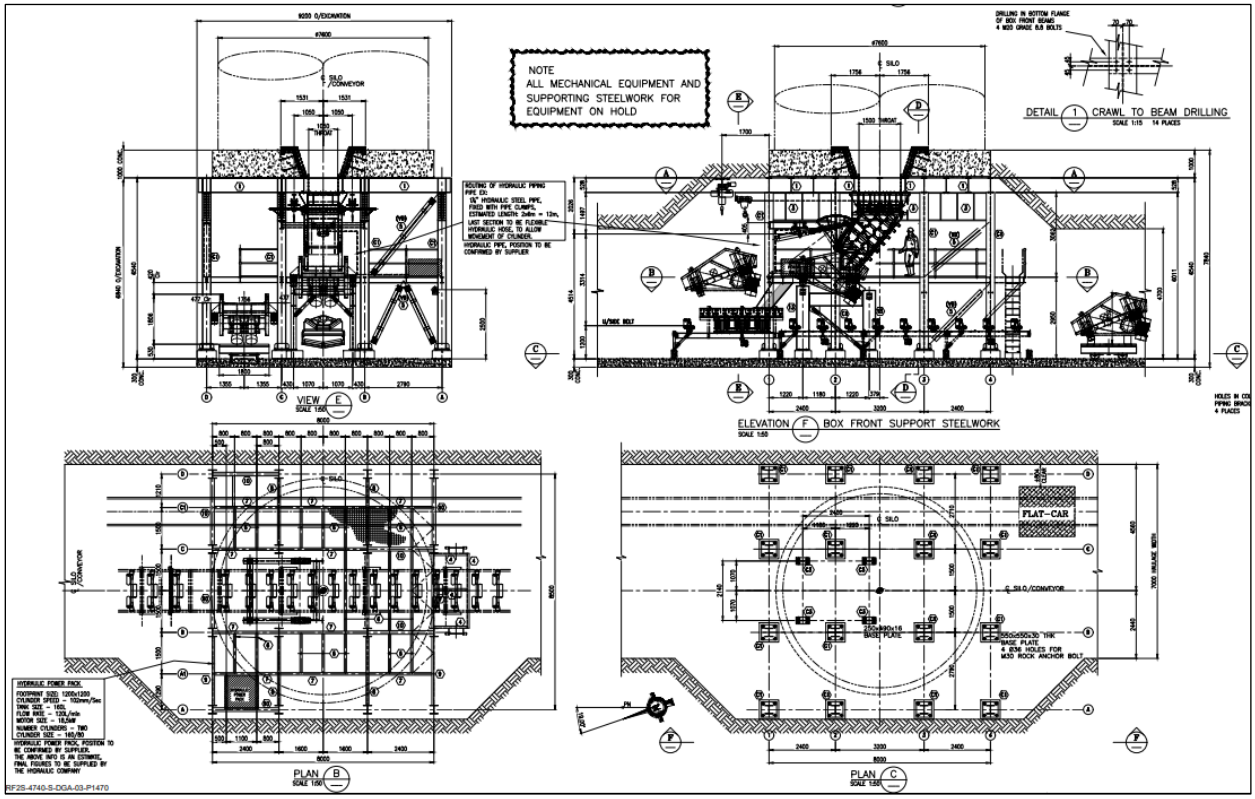
The skip loading will be conducted at the 285 lvl station.

Two fine ore bins each of 7.5 m diameter, 42 m height, and 2,750 t capacity will contain rock processed through the UG crusher and development muck passes. The fine ore bins will be collared at the 335 lvl Station and terminate at the 285 lvl Station, and feed directly into the shaft skip loading equipment. At full production (2.85 Mtpa) the fine ore bins will provide for 17 hours of storage capacity.

The fine ore bins shall be constructed using a combination of raise bore and/or drill and blast techniques, depending on the expertise of the adjudicated contractor.

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 16-52.

Figure 16-52: Fine Ore Bin Bulkhead



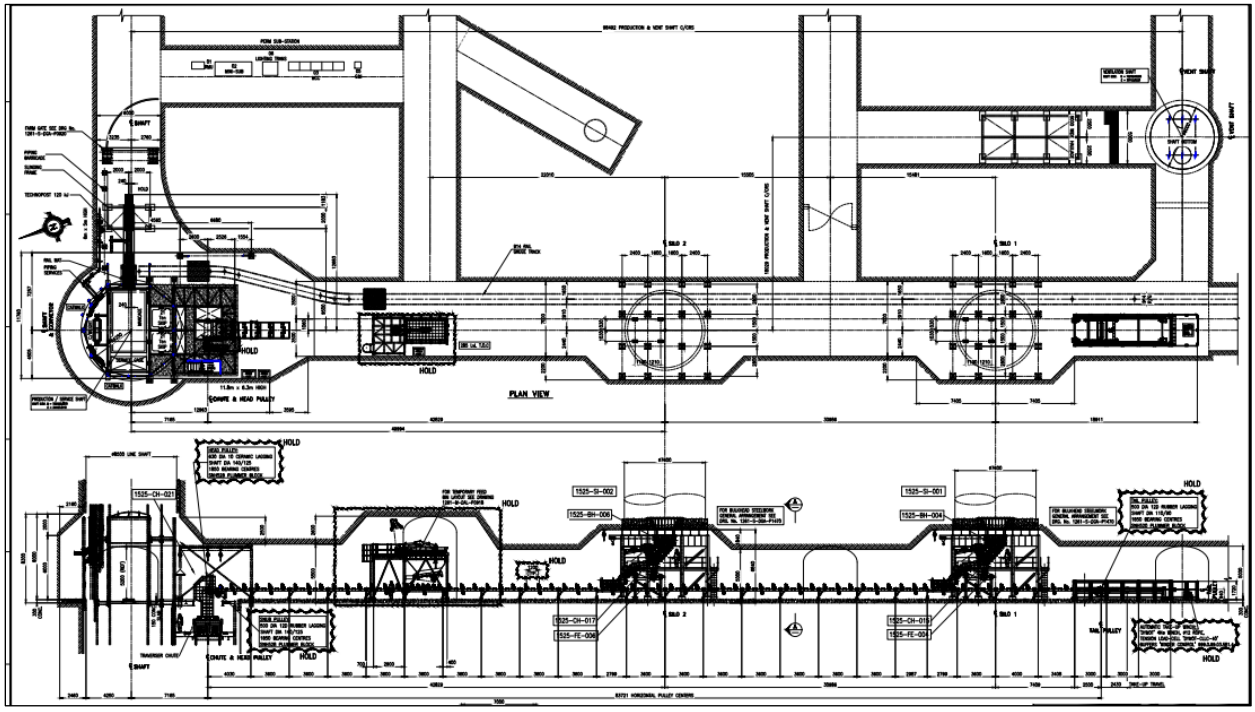
Source: UMS (2025)

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 accessible by ramp and shaft. The level will also serve as the primary dewatering pump station and access to shaft bottom and to the crusher chamber described in Section 16.8.1.

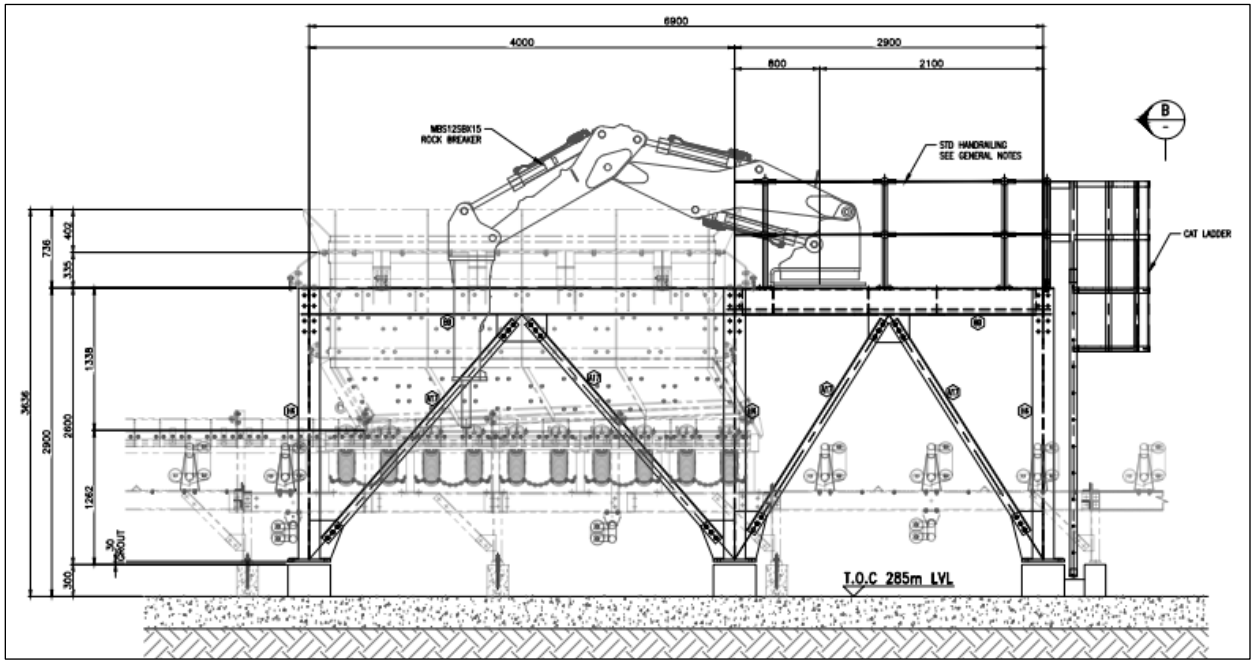
Figure 16-53: Skip Loading Station Feed Conveyor Layout



Source: UMS (2025)

The final configuration of the skip loading station will not be commissioned until after the underground crusher and conveyor is installed. In advance of this, a temporary skip loading conveyor and side loading bin will be installed immediately in front of the shaft loading pocket. This structure will receive run of mine waste rock via LHD manual loading and will be equipped with a static grizzly and rock breaker to control oversize as illustrated in Figure 16-54.

Figure 16-54: Temporary Feed Conveyor Side Loading

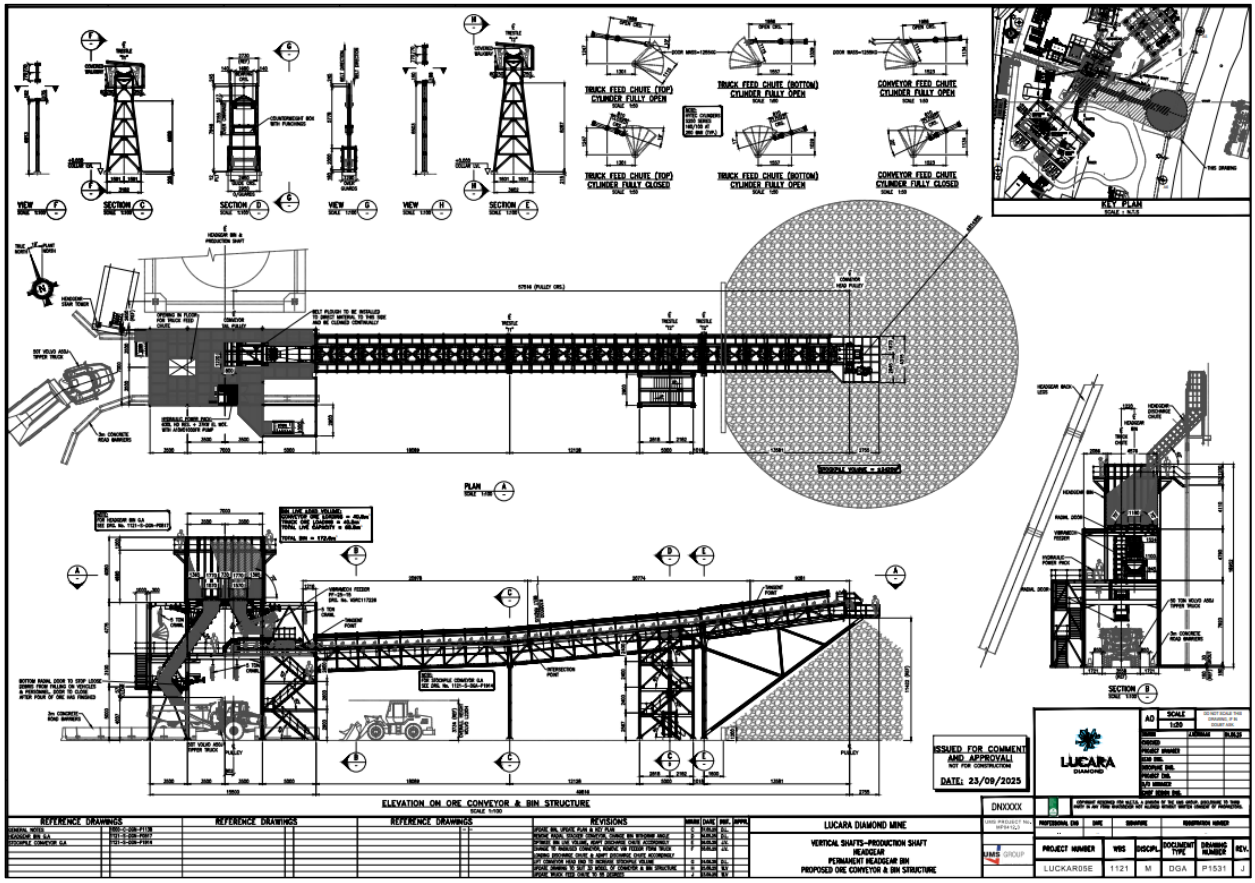


Source: UMS (2025)

16.7.2.7 Surface Discharge

At surface, hoisted materials will be tipped into a single P/S headgear bin which will provide a live capacity of 68.8 m^3. This bin will have two discharge chutes, one to a vibrating feeder and overland conveyor which reports to a 2,420 m^3 stockpile, or to a secondary truck loading chute capable of feeding 40 t trucks when the conveyor or stockpile is unavailable.

Figure 16-55: Headgear Bin and Conveyor



Source: UMS (2025)

16.7.2.8 Shaft Bottom

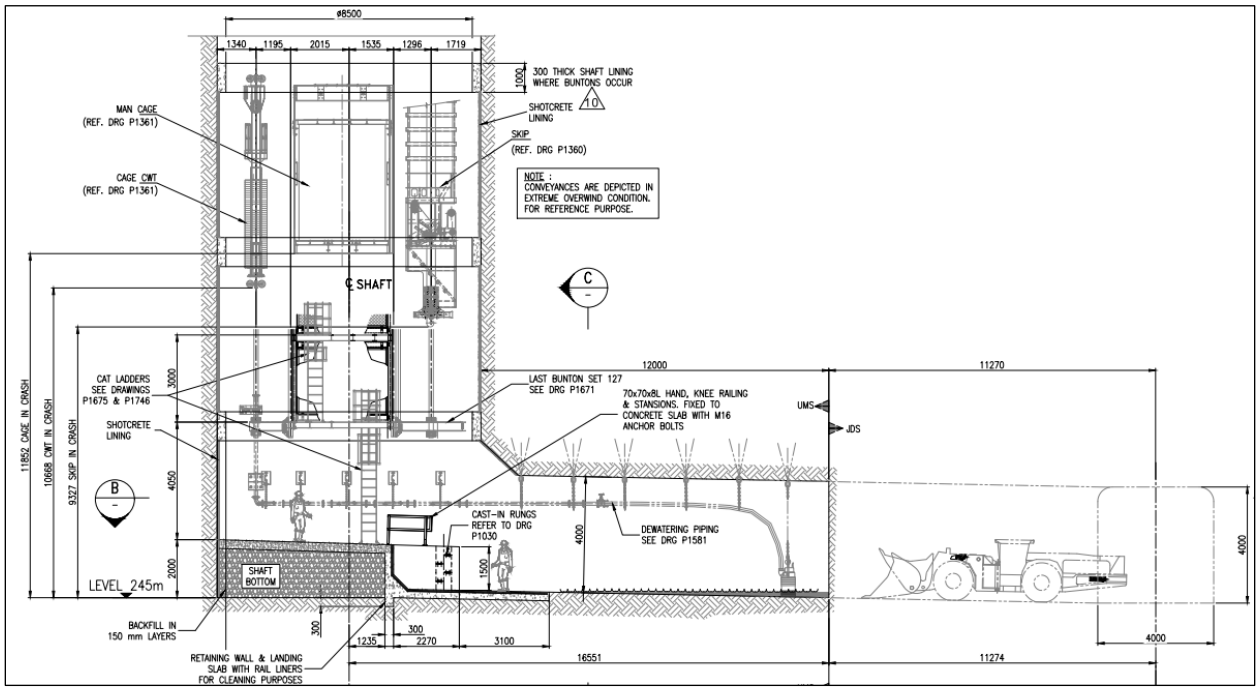
V/S Bottom is situated at the 285 lvl station. This station will serve as the primary return air route for the mine in permanent configuration. The shaft bottom will be equipped with a barricade to prevent unauthorised entry and concrete floor which is sloped to direct shaft inflows to a nearby sump.

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

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 side loading bin discussed in 16.7.2.6. 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 fixed ladder system 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. In permanent configuration a concrete barrier will prevent spillage handling LHDs from tramming too far into the shaft and exposing the operator to falling debris as shown in Figure 16-56.

Figure 16-56: Production Shaft Bottom



Source: UMS (2025)

16.7.3 Mine Levels

Table 16-17 outlines the mine levels, function, depths, and points of access. Levels are named after their UTM elevation. Refer to Figure 16-1 for an illustration of the mine levels.

Table 16-17: Mine Levels

Level (mL)	Function	Depth Below Surface	Level Spacing	Shaft Access
580	Cave Monitoring and Recovery	434	-	670
470	Stoping	544	110	470
405	Stoping	609	65	470
340	Undercut	674	65	310
310	Drawpoints	704	30	310
285	Crushing & Skip Loading, Dewatering	729	25	285
245	Shaft Bottom	729	40	285

16.7.3.1 Level Selection

The mine design is centralized around the extraction level, from which all production ore is extracted from the cave, and from which all other mine levels are offset.

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 maximise 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.

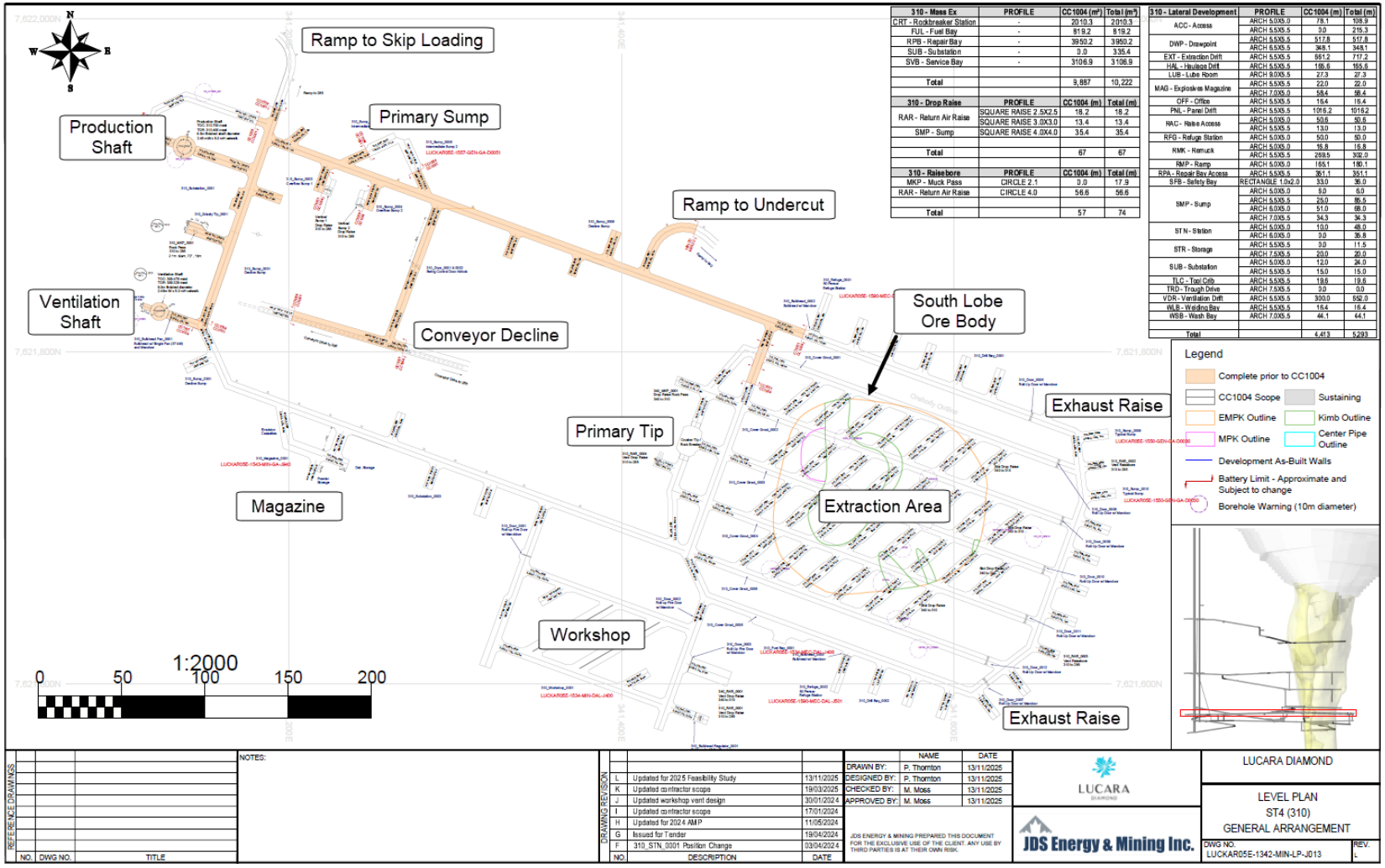
310 lvl was selected as the base extraction level in this exercise, as it had been previously identified as the most profitable extraction level in 2019 PCBC Footprint Finder evaluations (Doerksen Et al., 2019). The Arena simulations stepped the extraction level downwards in 10 m increments to the bottom of the indicated resource (250 masl).The results of the Arena simulation suggested that the then 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 masl given the simulation did not take into account geotechnical, hydrogeological, or operating risks. Given 310 masl offers 35% more drawpoints than 280 masl, it is anticipated that this 35% surplus will account for these risks. Furthermore, the reduced footprint at the lower elevations would likely jeopardise draw recovery of high value kimberlite on the margins of the pipe, given the tapered pipe geometry.

Further reviews of the extraction level were conducted in 2025 to address geotechnical concerns raised on the northwestern drawbells, and in light of revised cave simulation results. These reviews included raising the extraction level to 345 masl and reducing the amount of drill & blast required to initiate the cave. It was decided to hold the extraction level at 310 mL for negative impact to schedule, loss of early high grade production ore, and lack of geotechnical confidence the risks do not also exist at the higher elevation.

16.7.3.2 Extraction Level

The extraction horizon is located at 310 masl and is accessed from the 310 lvl P/S shaft station.

Figure 16-57: 310 lvl Plan View



310 - Mass Ex	PROFILE	CC1004 (m³)	Total (m³)
CRT - Rockbreaker Station	-	2010.3	2010.3
FUL - Fuel Bay	-	819.2	819.2
RPB - Repair Bay	-	3950.2	3950.2
SUB - Substation	-	0.0	335.4
SVB - Service Bay	-	3106.9	3106.9
Total		9,887	10,222

310 - Drop Raise	PROFILE	CC1004 (m)	Total (m)
RAR - Return Air Raise	SQUARE RAISE 2.5X2.5	18.2	18.2
	SQUARE RAISE 3.0X3.0	13.4	13.4
SMP - Sump	SQUARE RAISE 4.0X4.0	35.4	35.4
Total		67	67

310 - Raisebore	PROFILE	CC1004 (m)	Total (m)
MKP - Muck Pass	CIRCLE 2.1	0.0	17.9
RAR - Return Air Raise	CIRCLE 4.0	56.6	56.6
Total		57	74

310 - Lateral Development	PROFILE	CC1004 (m)	Total (m)
ACC - Access	ARCH 5.0X5.0	78.1	108.9
	ARCH 5.5X5.5	0.0	215.3
DWP - Drawpoint	ARCH 5.5X5.5	517.8	517.8
EXT - Extraction Drift	ARCH 5.5X5.5	348.1	348.1
HAL - Haulage Drift	ARCH 5.5X5.5	661.2	717.2
LUB - Lube Room	ARCH 9.0X5.5	165.6	165.6
MAG - Explosives Magazine	ARCH 5.5X5.5	27.3	27.3
OFF - Office	ARCH 7.0X5.0	22.0	22.0
PNL - Panel Drift	ARCH 5.5X5.5	58.4	58.4
RAC - Raise Access	ARCH 5.5X5.5	1016.2	1016.2
RFG - Refuge Station	ARCH 5.0X5.0	50.6	50.6
RMK - Remuck	ARCH 5.0X5.0	13.0	13.0
RMP - Ramp	ARCH 5.0X5.0	53.0	53.0
RPA - Repair Bay Access	ARCH 5.5X5.5	15.8	15.8
SFB - Safety Bay	ARCH 5.5X5.5	269.5	302.0
SMP - Sump	RECTANGLE 1.0x2.0	33.0	36.0
	ARCH 5.0X5.0	5.0	5.0
	ARCH 5.5X5.5	25.0	85.5
	ARCH 6.0X5.0	51.0	68.0
	ARCH 7.0X5.5	34.3	34.3
STN - Station	ARCH 5.0X5.0	10.0	48.0
	ARCH 6.0X5.0	0.0	35.8
STR - Storage	ARCH 5.5X5.5	0.0	11.5
	ARCH 7.5X5.5	20.0	20.0
SUB - Substation	ARCH 5.0X5.0	12.0	24.0
	ARCH 5.5X5.5	15.0	15.0
TLC - Tool Crib	ARCH 5.5X5.5	19.6	19.6
TRD - Trough Drive	ARCH 7.5X5.5	0.0	0.0
VDR - Ventilation Drift	ARCH 5.5X5.5	303.0	652.0
WLB - Welding Bay	ARCH 5.5X5.5	16.4	16.4
WSB - Wash Bay	ARCH 7.0X5.5	44.1	44.1
Total		4,413	5,293

REFERENCE DRAWINGS

NO.	DWG NO.	TITLE

NOTES:

DRAWING REVISION

NO.	DESCRIPTION	DATE
L	Updated for 2025 Feasibility Study	13/11/2025
K	Updated contractor scope	19/03/2025
J	Updated workshop vent design	30/01/2024
I	Updated contractor scope	17/01/2024
H	Updated for 2024 AMP	11/05/2024
G	Issued for Tender	19/04/2024
F	310_STN_0001 Position Change	03/04/2024

	NAME	DATE
DRAWN BY:	P. Thornton	13/11/2025
DESIGNED BY:	P. Thornton	13/11/2025
CHECKED BY:	M. Moss	13/11/2025
APPROVED BY:	M. Moss	13/11/2025

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LUCARA DIAMOND

JDS Energy & Mining Inc.

LEVEL PLAN
ST4 (310)
GENERAL ARRANGEMENT

DWG NO. LUCKAR05E-1342-MIN-LP-J013

REV. L

The 310 lvl will remain active for the LOM and provide access to the following infrastructure:

- Drawpoints and Primary Tip;
- Crusher Conveyor;
- Workshop and Warehouse;
- Magazine;
- Permanent Refuge;
- Primary Sumps;
- Ramp to 340 lvl Undercut; and
- Ramp to 285 lvl Skip Loading Station.

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 lvl 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 dual drives will also assist with separating pedestrian and trackless machinery haulage in permanent configuration.

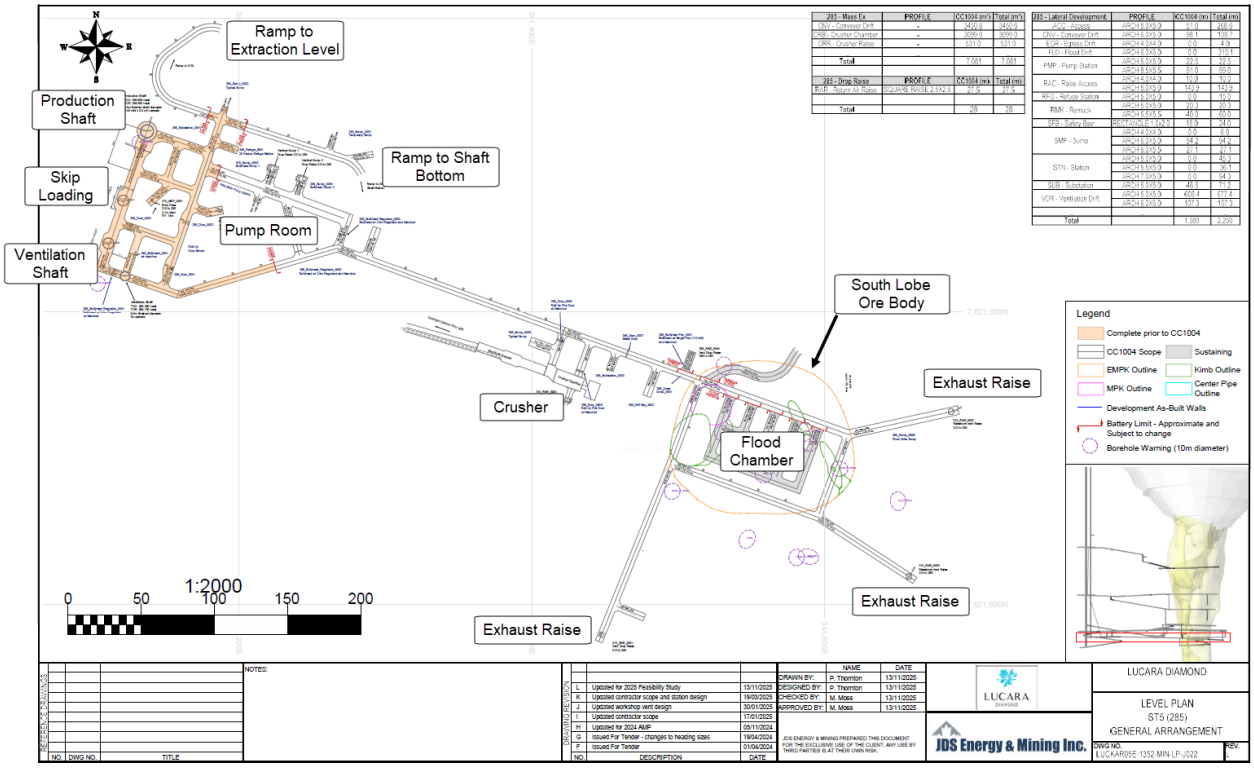
Development will incline from the shafts up to the start of the extraction area to direct all water produced during development back towards the shaft station sump. Within the Extraction perimeter drive, development will grade away from the shafts to the east and towards a pair of sumps dedicated to managing groundwater inflows produced by the cave.

Waste rock generated through the development of the 310 lvl will be trammed towards the V/S and tipped down a rock pass equipped with static grizzly fixed breaker. This pass will report to the skip loading station where another LHD will rehandle material onto a side loading conveyor illustrated in Figure 16-54. Once the UG crushing and conveying system is commissioned, development waste will report to the primary tip.

16.7.3.3 Services Horizon

The Services horizon is located at 285 lvl and is accessed from the 285 mL P/S shaft station. Figure 16-58 shows a plan view of the 285 lvl.

Figure 16-58: 285 lvl Plan View



The 285 lvl will remain active for the LOM 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 lvl will be accessed via the P/S, as well as by ramp from the 310 lvl. A single drive will extend from the shaft station underneath of the 310 mL extraction area. Ventilation raises will be driven from the 310 lvl to the 285 mL 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 16.7.7.4.

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.

16.7.3.4 Drilling Horizons

Above the Extraction Level, three (3) production drill & blast horizons will facilitate assisted caving up to 495 masl. One Undercut level and two stoping levels.

The undercut level, discussed further in Section 16.7.4.1, is situated 15 m above the extraction drives to maintain safe working sill pillars during mine construction.

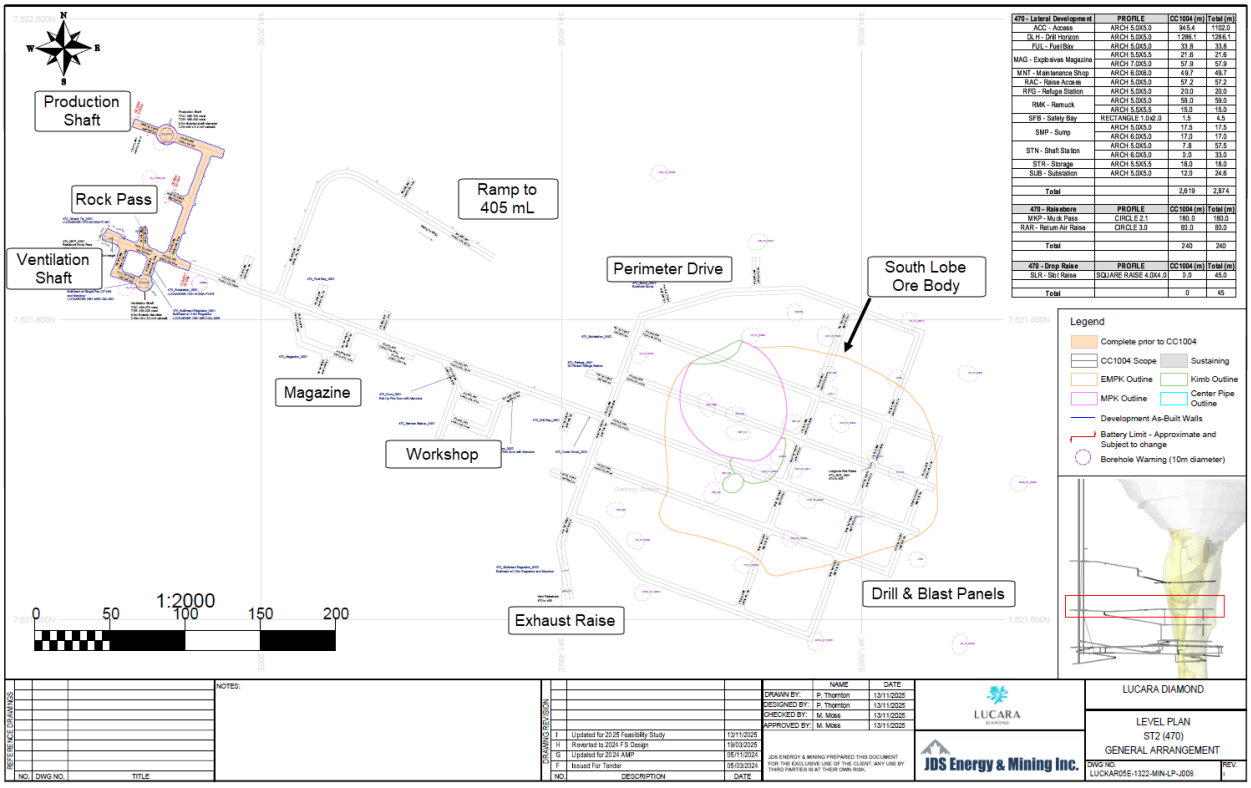
Stoping levels are spaced 65 m apart to minimise lateral development requirements, work within effective drill limitations, and terminate at the elevation in which the South Lobe is expected to start caving naturally.

Drilling horizons will be accessed by shaft station or ramp from another level. They 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, 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 31.5 m spacing across the South Lobe. A perimeter drive will be driven 30 m offset from the ore body and a crosscut will be driven mid-point across the ore body perpendicular to the drill drives. This additional development will serve as secondary egress and provide multiple points of access and working faces for production drills and charging crews.

Level infrastructure will be established on the footwall or perimeter drive, serviced by the nearest shaft station. Figure 16-59 illustrates a typical drill horizon.

Figure 16-59: 470 lvl Drill Horizon Plan View



Waste rock generated through development of the 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 rock breaker to reduce oversize to a minimum 300 mm passing. LHDs will rehandle muck between passes until reporting either to the primary crusher tip or side loading skip conveyor bin.

Waste passes will be developed by raisebore. Requirements for raise support or grouting will be subject to evaluation closer to execution.

Fresh air will be supplied to the drill horizons by the P/S, with return air descending to the 285 lvl main return air drive via a series of ventilation raises located at the footwall drive of the South Lobe. Raises will be 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 lvl undercut, and the last is the 470 lvl production horizon. Drill horizons will be temporary in nature and closed off with an air blast barricade 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.

16.7.3.5 Flood Chamber

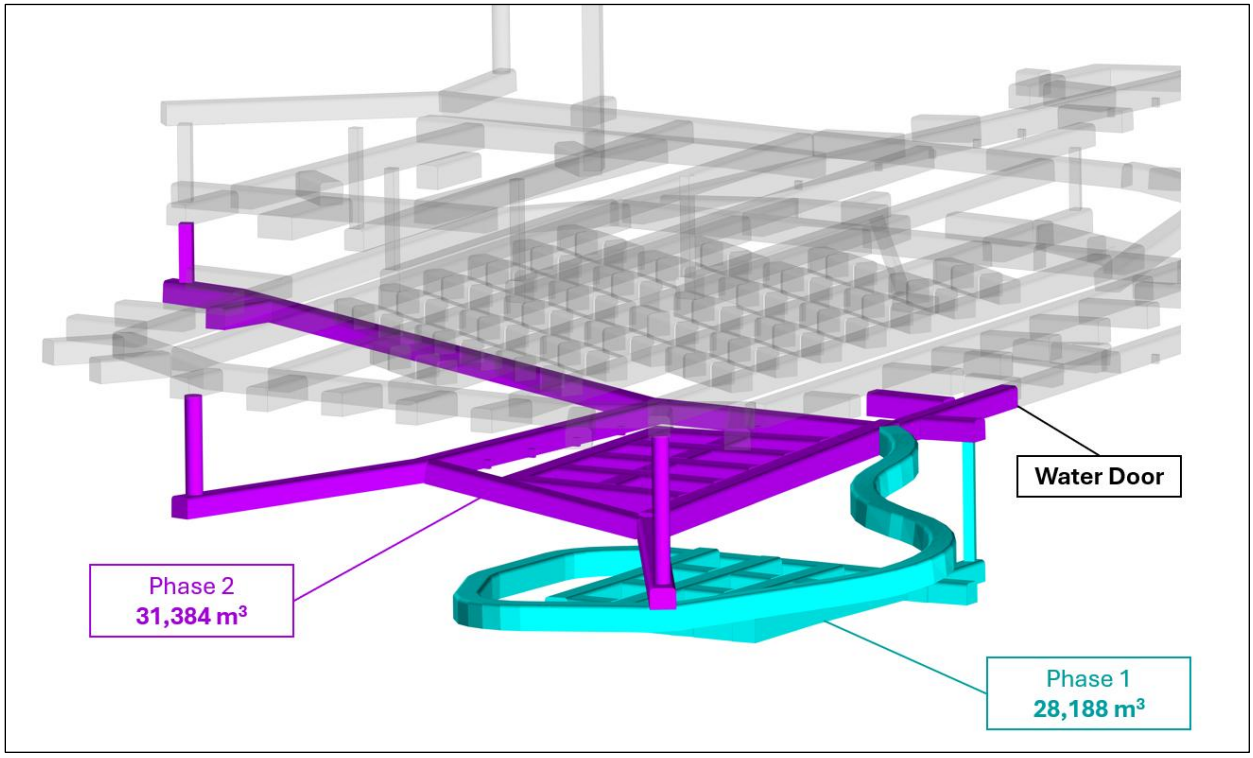
Development will grade towards the shafts up to the entry of the crusher chamber in an effort to report all water produced during development back towards the shaft station sump. Beyond this point development will grade towards the ore body and, when pumping systems are overwhelmed, overflow into a flood chamber sized for 59,600 m^3 capacity. This flood chamber, further described in Section 16.7.7.4, will serve dual purpose as the primary return air circuit for the mine.

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 water-tight door will be installed west of 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.

Figure 16-60 illustrates the flood chamber capacity, and location of water-tight door. In a flooding event Phase 1 will allow water to collect in the lower level of the flood drive while operations continue as normal. Once full, water will begin filling rooms on the upper level in Phase 2 until ventilation circuits become compromised and water levels reach the crusher entry. At this point the water-tight door will be closed and water will be allowed to fill the higher elevation drives and ultimately back up the ventilation raises to the extraction level. Refer to Section 16.7.7.4 and the process flow diagrams within for details on the dewatering systems that will manage these events. Should additional stormwater storage be required there is opportunity to stope out vertical ore chambers between the upper and lower rooms, subject to geotechnical investigation.

Figure 16-60: 250 lvl Flood Chamber



16.7.3.6 Crusher and Conveyor Levels

The crusher is located below the 310 lvl extraction area and will have two primary points of access:

1. Bottom entry via the 285 lvl return air drive; and

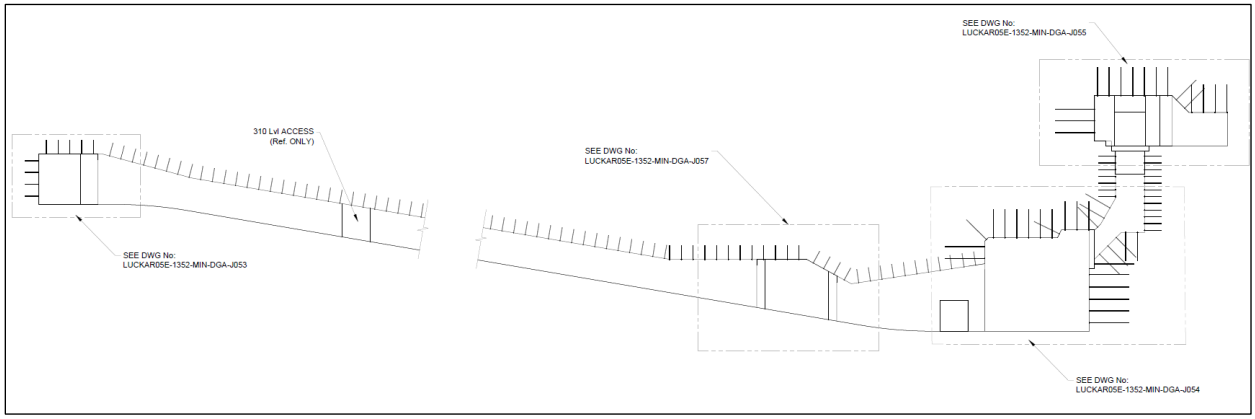
2. Top entry via the conveyor drive, which has an access point on the 310 lvl.

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 lvl 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 lvl, which will provide all services to the crusher chamber.

Bottom entry to the crusher chamber from the 285 lvl 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 ventilated from a drop raise driven adjacent to the primary tip.

Figure 16-61: UG Crusher and Conveyor Excavation Layout



Source: JDS (2024)

16.7.3.7 Cave Recovery Level

The mine design includes underground development 85 m above the final stoping horizon at 495 masl and 85 m below the pit floor, providing drill vantage for preconditioning, re-initiating, or recovering portions of the cave. This development will be driven from the 670 lvl shaft station and consists of a 90 m vertical ramp to the 580 lvl. A footwall drive offset 30 m from the ore body will house level infrastructure. A single crosscut will be driven through the ore body to provide drill vantage. This cavity monitoring will complement the majority of the cave monitoring instrumentation which will be conducted in surface drillholes. There is opportunity to install additional cave monitoring instruments from this level to reduce cost of drilling from surface.

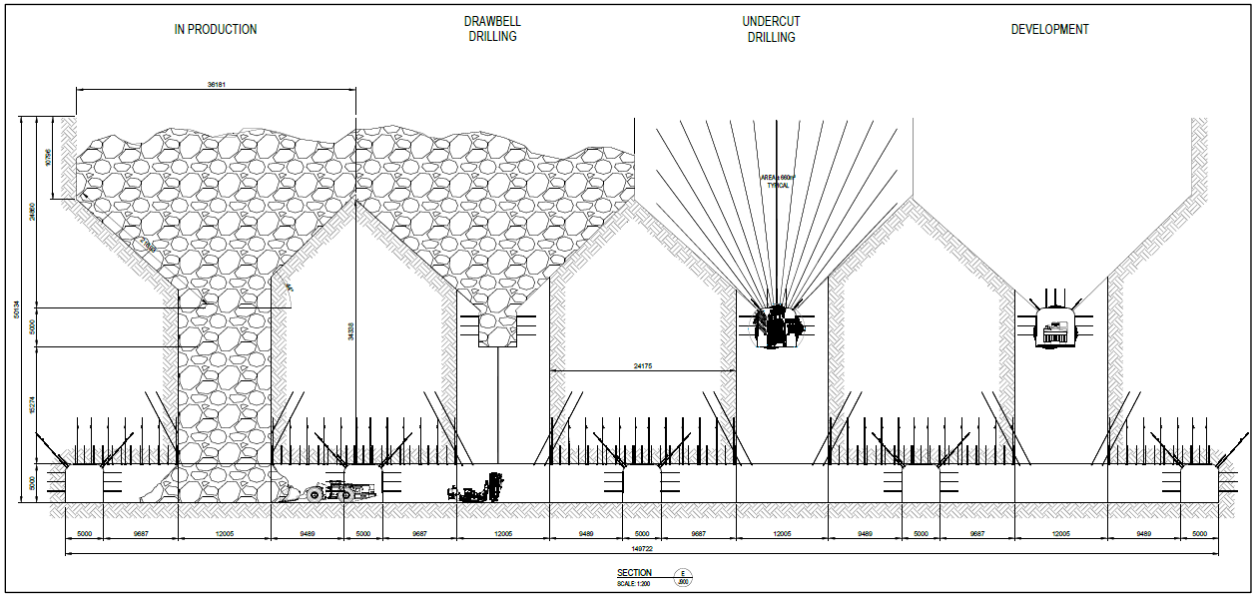
For further details on the cave monitoring strategy and design, refer Section 16.3.14.

16.7.4 Drawbells and Undercut

Drawpoints will be established to access drawbells, using an El Teniente layout selected for improved pillar stability and reliability in challenging ground conditions. Since drawbells are commonly utilised in conventional caving operations, this method enables the project to leverage established industry knowledge in both design and planning.

Drawbells will be created with upholes from the drawpoints and blasted into the blasted undercut (Advanced Undercut) above, as shown in Figure 16-62.

Figure 16-62: Drawbell and Advanced Undercut Design



Major and minor apex pillar dimensions as stated in Table 16-18 follow industry guidelines and maximise size while ensuring enough drawpoints for production targets. Drawpoints are illustrated in Figure 16-63 and Figure 16-64.

Table 16-18: Karowe Drawpoint Pillar Dimensions

Drawpoint Pillar Design Parameters	Units	Dimensions
Primary Apex Pillar Dimensions (W x H)	m	24 x 32
Secondary Apex Pillar Dimensions (W x H)	m	13 x 21

Figure 16-63: El Teniente Drawpoint Arrangement

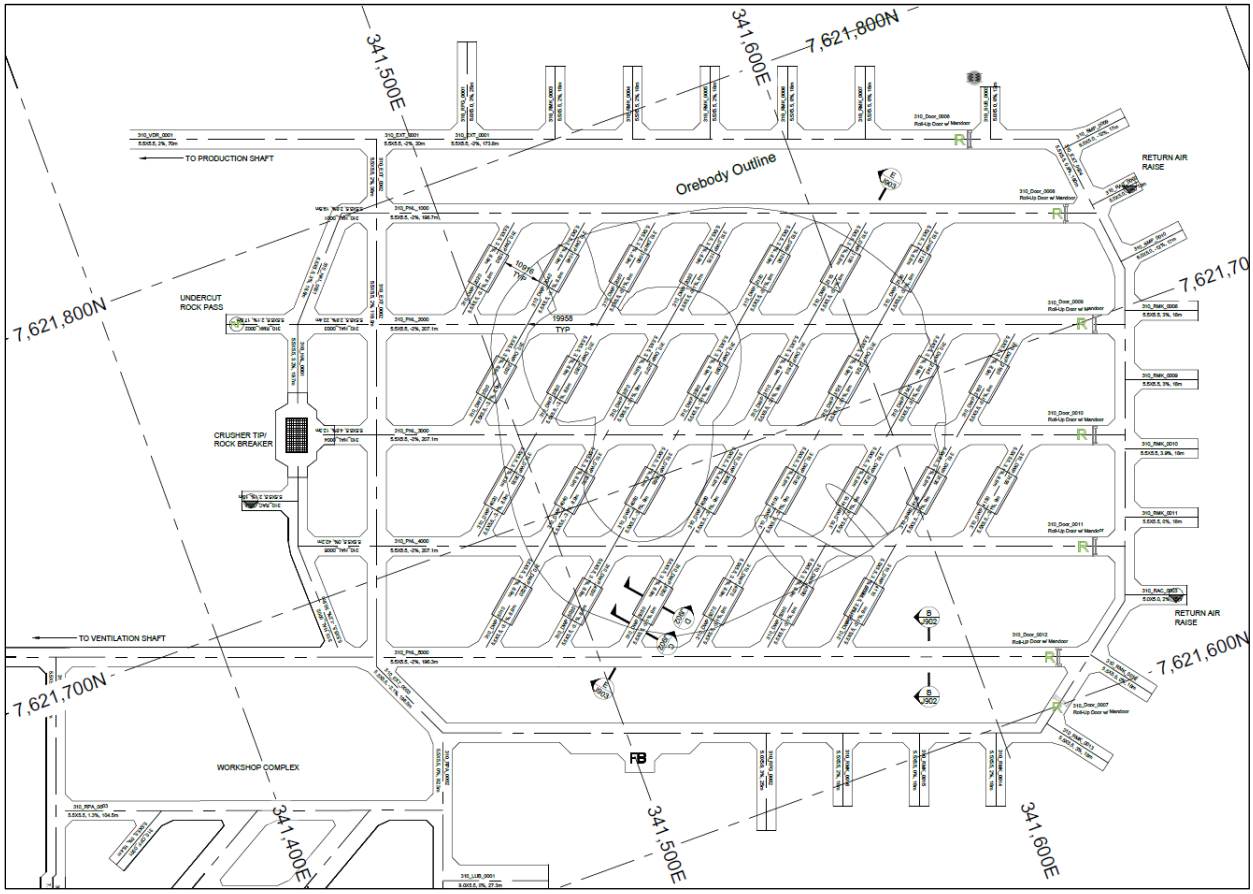
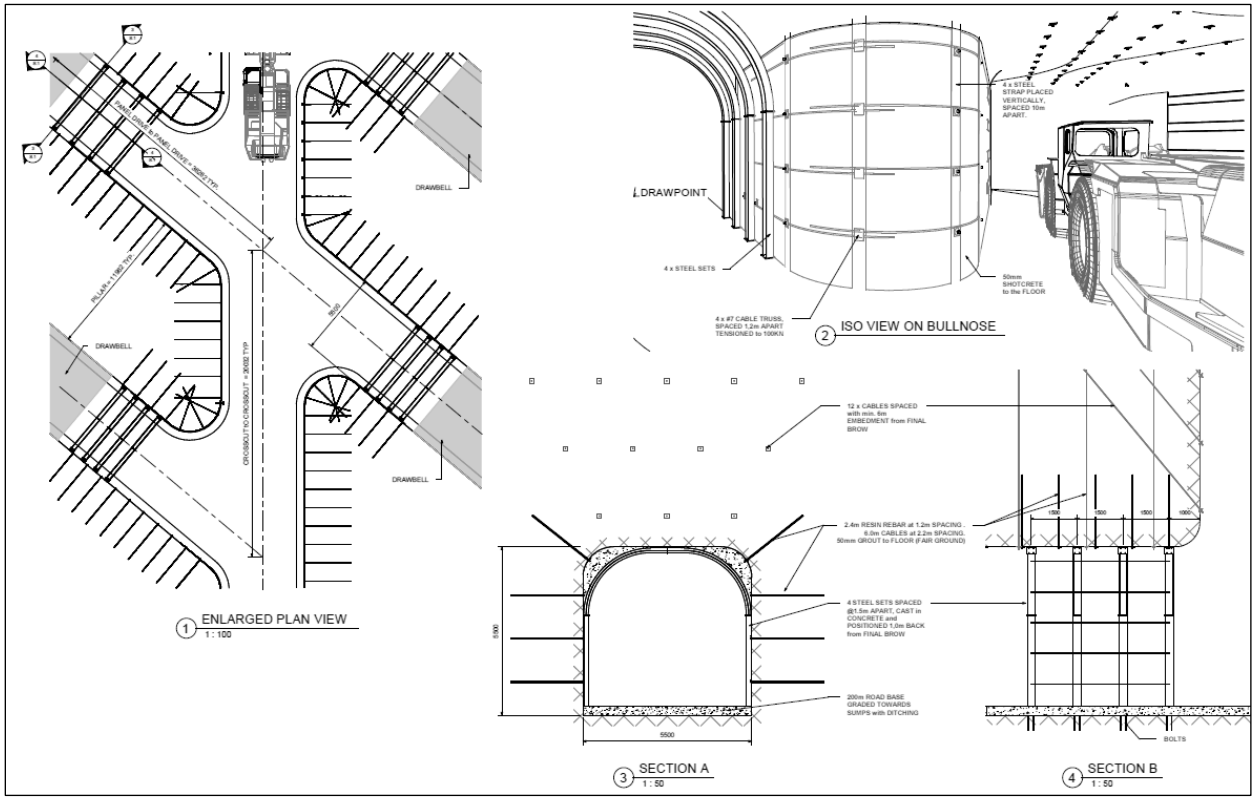
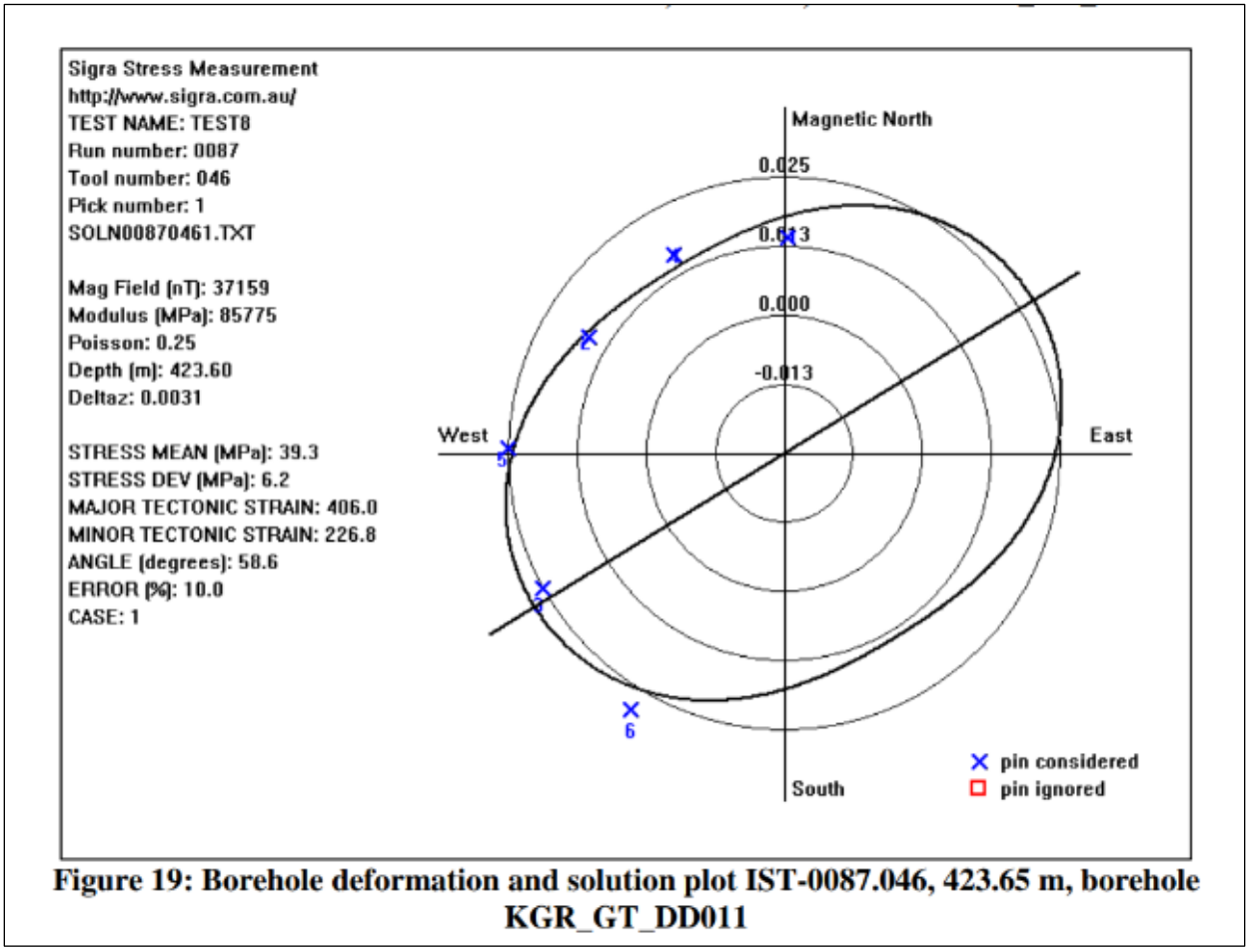


Figure 16-64: Drawpoint Excavation Details



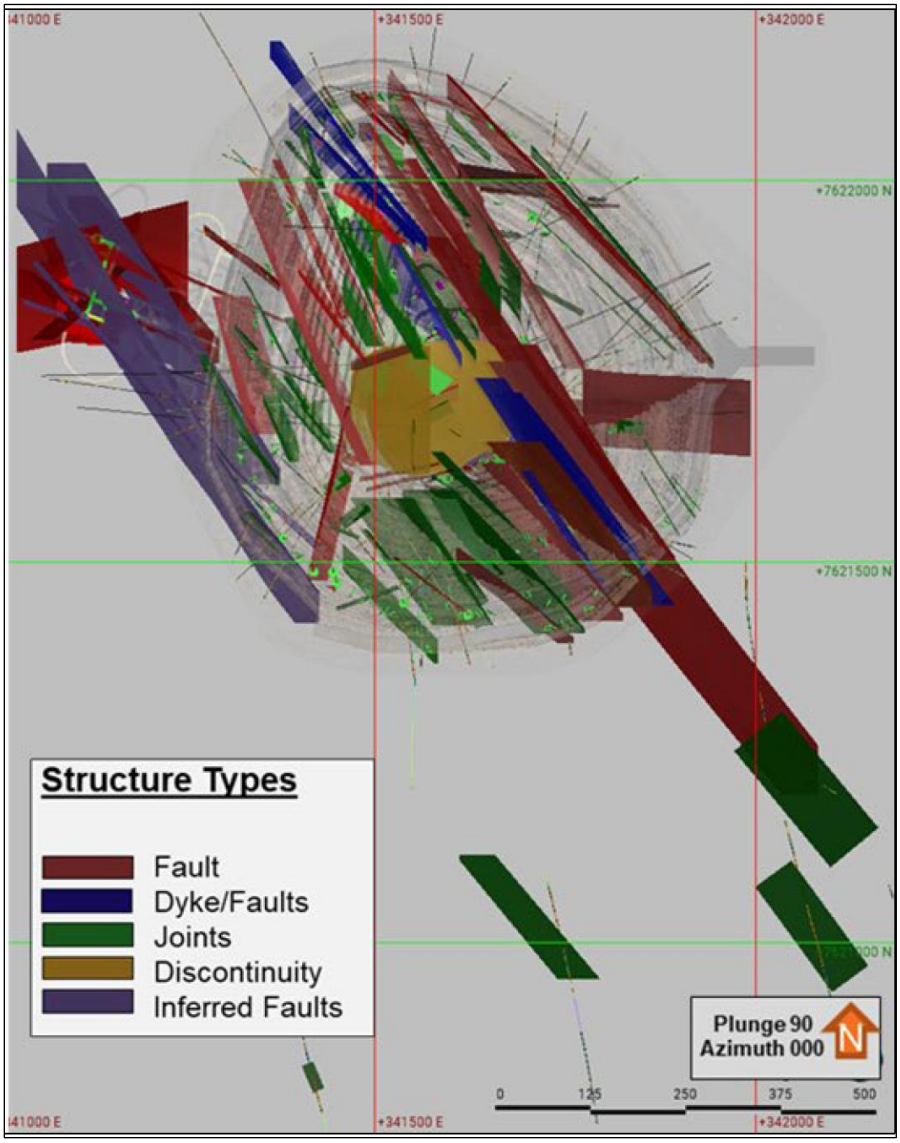
Drawpoints and undercuts are oriented at shallow angles to the major principal stress direction, and normal to prevailing major structures (striking NNW and NE), to reduce the risks of adverse ground behaviour (see Figure 16-66). Blasting of the undercut rings is sequenced from the ENE towards the WSW, with a minimal lead ahead of the main long-hole rings above.

Figure 16-65: Karowe Sigra™ Stress Measurements



Source: Sigra™ (2019), as contained in the Lucara Ground Control Management Plan (Middindi, 2025)

Figure 16-66: Karowe Main Structural Feature Orientations



Source: SRK (2024)

16.7.4.1 Advanced Undercut

In conventional block caving the undercut creates the initial void above the drawbells, over the required footprint, to trigger cave initiation. Undercuts may be blasted as post or advanced undercuts, depending on the relative timing in relation to excavation of the drawbells. In high stress mining environments, advanced undercuts are typically favored. This approach enables the advancing undercut face to absorb the high abutment stresses and allow the safe excavation and construction of drawbells in a low-stress setting.

The South Lobe is not situated in a high stress environment, and an advanced undercut is not stipulated for geotechnical reasons. However, it has been planned as an advanced undercut to maximise the safe and stable construction of drawbells and extraction level pillars. Standoff distances between the undercut and bell establishment will maintain a 45-degree stress shadow, or at minimum 20 m.

An added benefit to the advanced undercut method is the ability to start large scale drill and blast operations early and bring ore forward in the production schedule. In natural cave mines the cave only begins to propagate once the undercut opens a span that promotes instability of the cave. In a post-undercut method this action waits for the drawbells to be constructed which can be lengthy, especially in weak ground where drawpoint construction requires steel sets and other structural supports.

Undercuts are often of limited vertical extent to achieve the undercutting objective whilst minimizing the tonnes to be removed. Karowe's geomechanical model shows the cave back stays stable for more than 170 m above the extraction level, allowing large blast rings to be used in the undercut to boost initial ore output.

The undercut level will be at 340 masl and include the following features:

- Positioned 15 m above the draw points;
- Consists of four (4) crosscuts spaced 31.5 m apart;
- Includes a footwall drive 30 m West of the orebody;
- Equipped with rock pass to the primary tip;
- Ventilated via dedicated exhaust connection to the 285 RAR drive; and
- Serviced with refuge chamber, substation, sump, and other on-level infrastructure.

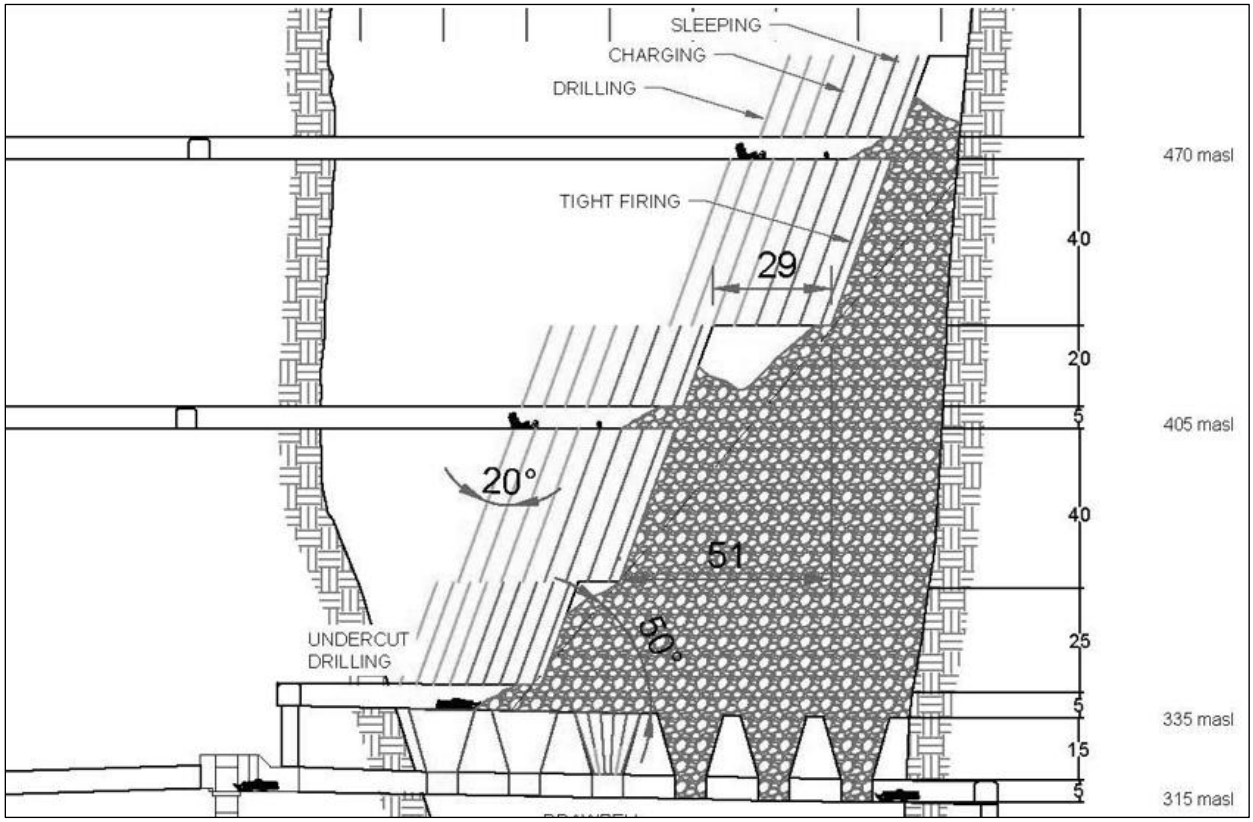
16.7.5 Blast Assisted Caving

16.7.5.1 Stope Design

Blast assisted stoping will be from the top of the undercut up to 495 masl. Stopes have been designed to minimise capital development requirements of sub levels while achieving high degrees of drillhole accuracy.

Stopes will be mined in 65.0 m x 31.5 x 5.0 m rings in a retreat fashion. Two stoping levels, each with four parallel stopes, will be mined from east to west in an en echelon pattern to target a stable arched cave back profile. Dump angles of 20 degrees and ring offsets between blasting horizons using both down- and uphole drilling will achieve a toe-to-toe blasting front of approximately 50 degrees as illustrated in Figure 16-67.

Figure 16-67: Stope Sequencing General Arrangement

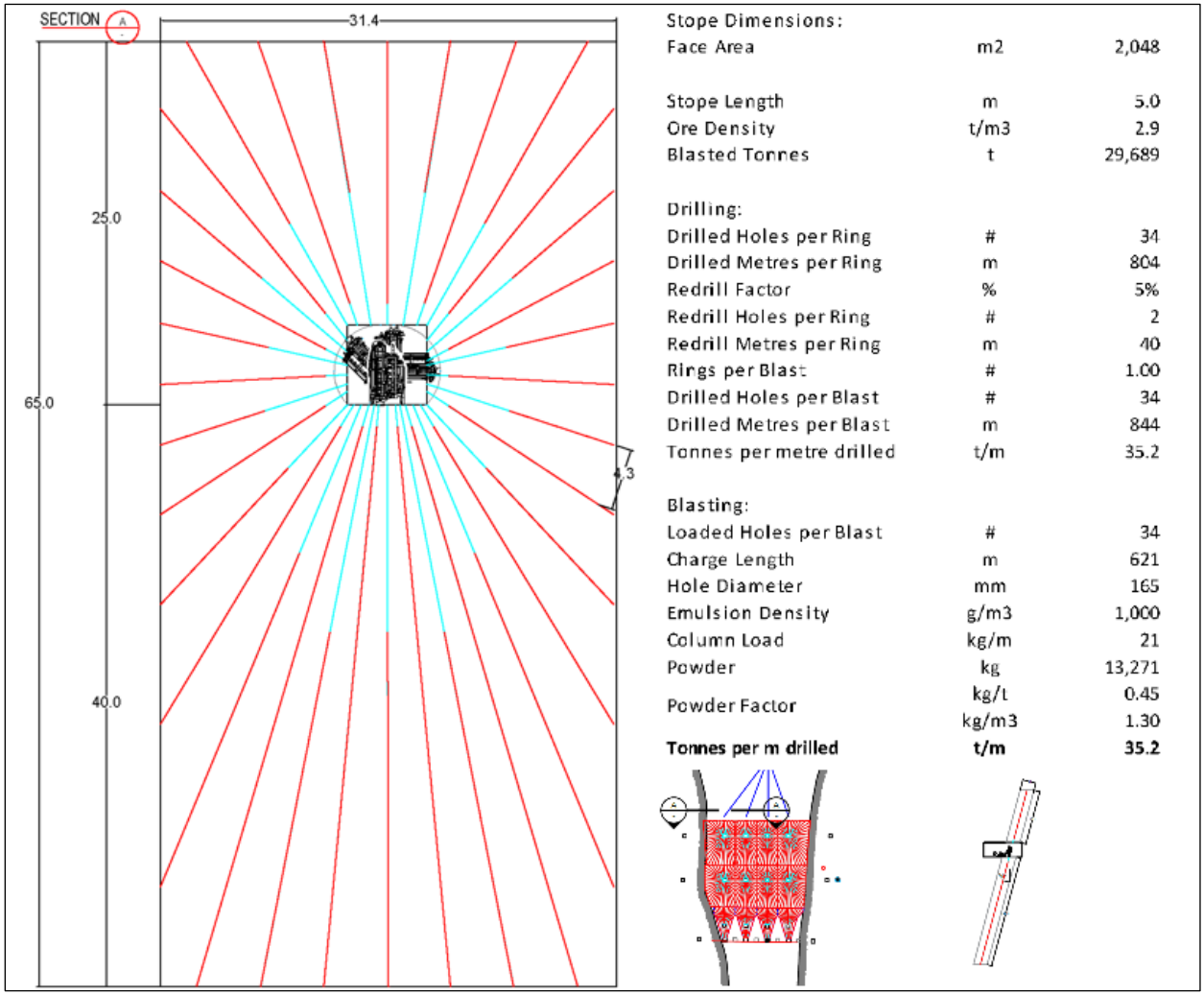


16.7.5.2 Production Drill Pattern

The KDM OP utilises a 0.3 - 0.4 kg/t powder factor and achieves excellent fragmentation with more than 90% passing 400 mm with the use of 165 mm blast holes on a 3.50 m x 4.50 m blast pattern (Lucara, 2021). UG stope drilling will be designed to achieve similar fragmentation with considerations for secondary fragmentation in the cave.

Stope drilling will be performed using a combination of 40 m down holes and 25 m up holes drilled via a series of 5.0 m x 5.0 m drill drives developed transversely through the ore body. Blast holes up to 165 mm diameter will be drilled by an In-The-Hole (ITH) hammer production drill on a 4.35 m x 5.00 m pattern. Average length of hole per 65 m tall stope will be 24 m with an average 35 t/m drilled. Refer to Figure 16-68.

Figure 16-68: Stope Design General Arrangement



Each ring will contain 34 blast holes drilled radially. Wireless detonators will allow rings to be charged and slept for up to three months before the blast and eliminate the need for charging crews to work near the open cave.

Stopes will be tight-fired into the cave against previously blasted material. Ring firing will be limited to one (1) ring at a time to minimise draw requirements and risk of exposing the blast front to the open cave back.

Stopes will be charged with 13,300 kg of pumpable gassed emulsion with density 1,000 g/cm^3 to achieve a powder factor of 1.3 kg/m^3.

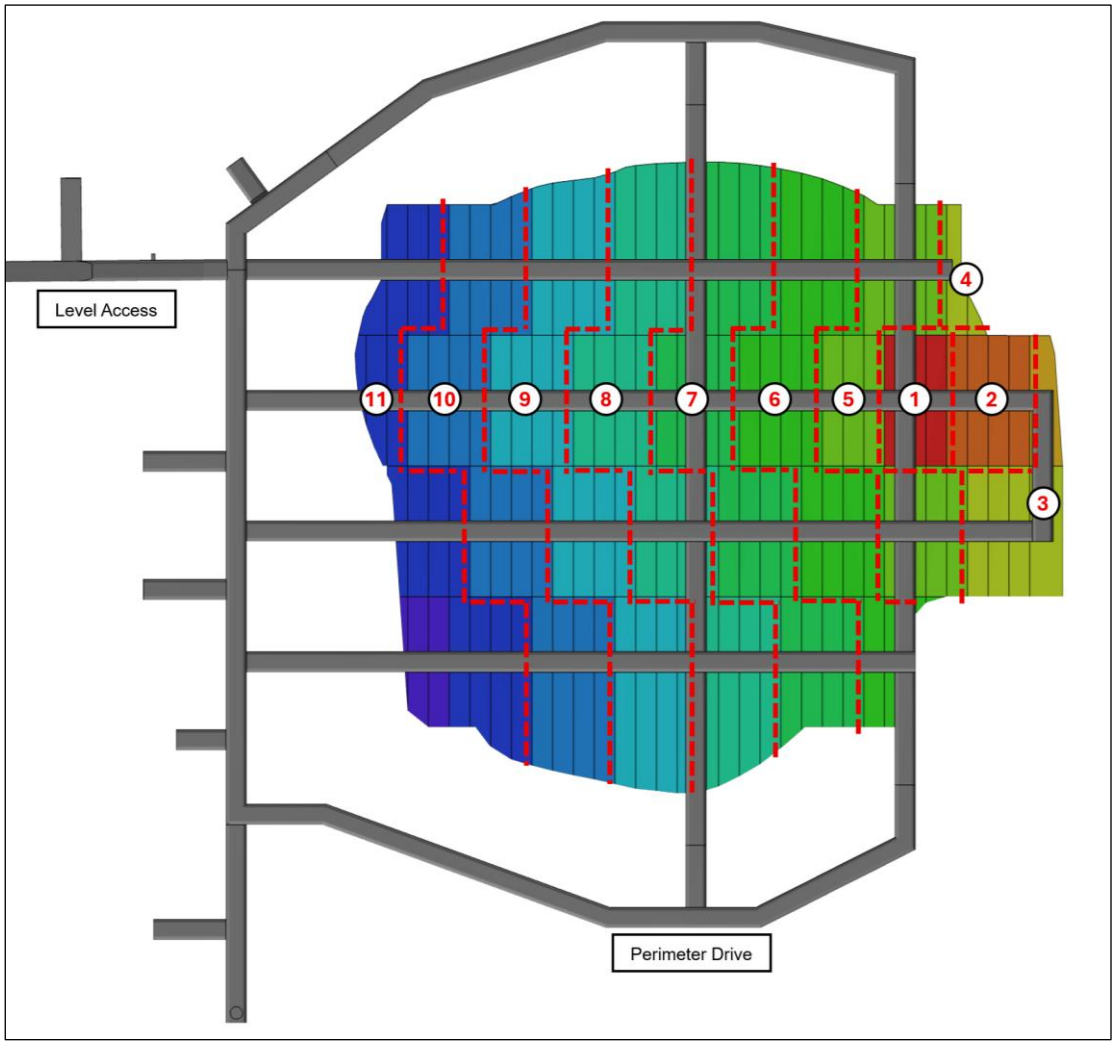
16.7.5.3 Stope Modelling

Stopes were designed using Deswik 3D Mine Planning software by slicing and appending an undiluted resource wireframe without stope optimization tools. They ensure vertical continuity to drawpoints, occasionally extending into waste to maintain this connection.

16.7.5.4 Stope Sequencing

A slot raise will provide the initial blast void and free face for the longhole stopes to break into. A slot drive will be driven across the width of the stope at the eastern side of the South Lobe from which a V30 slot will be drilled and slashed out. Long-hole stopes will be drilled and blasted in retreat fashion from east to west as illustrated in Figure 16-69.

Figure 16-69: Plan View of Typical Blasting Sequence



16.7.6 Natural Caving

16.7.6.1 Cave Design

Natural caving will be from 495 masl to surface via the OP. Cave assessments, strategy, and propagation is discussed in Sections 16.3.8 and 16.3.9 respectively.

16.7.6.2 Fragmentation

Fragmentation will be achieved through both blasting and natural comminution during material draw. The four stages of fragmentation at Karowe, from drawpoint establishment to steady state production are defined as:

Category 1: Blasted ore from Drawbells and Undercut.

Category 2: Blast assisted caving directly above the Undercut. Overhand stoping to pre-condition the 150 m base of the ore column.

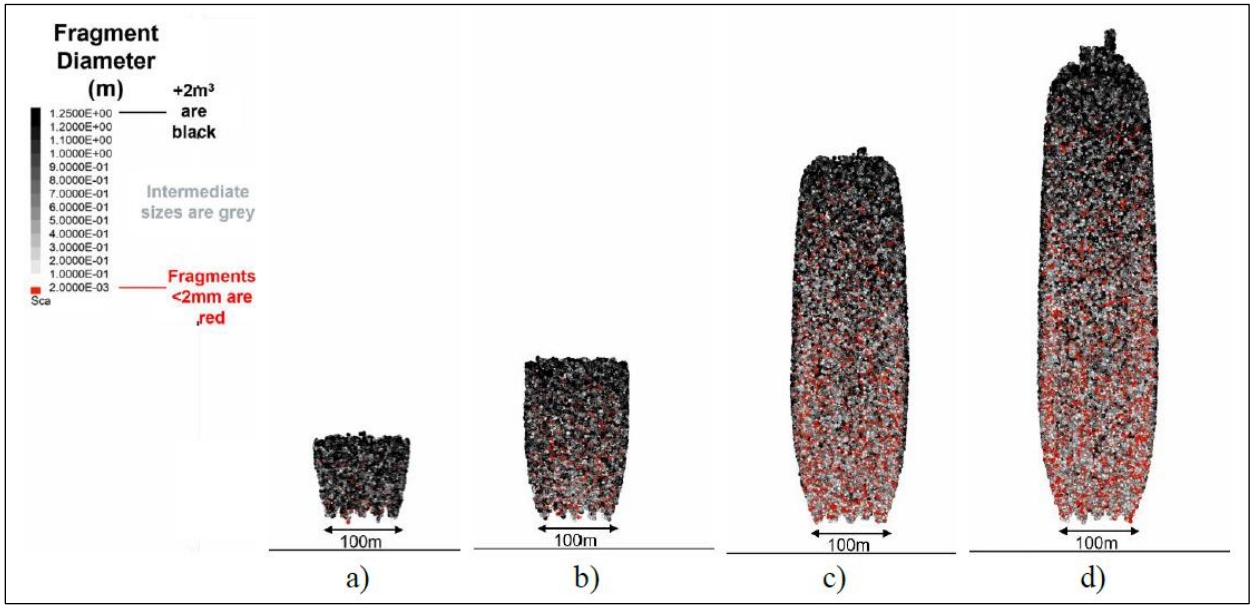
Category 3: Blocks formed during natural caving.

Category 4: Secondary fragmentation from material flow through the cave and into the drawpoints.

Categories 1 and 2 will be addressed using drill and blast control measures. The blast designs are informed by over thirteen years of on-site refinement of drill and blast practices, which typically result in more than 90% passing 400 mm (Lucara, 2021). Detailed engineering will include oversight and input from third party drill-and-blast experts, including engineering support from suppliers of wireless detonation technologies, which extends into the execution phase.

At the onset of natural caving (Category 3), the cave will initially generate relatively coarse material, with block size informed by the stress path induced by the blasting below, and the frequency and orientations of natural fractures. As these coarse materials move toward the drawpoints and interact with one another they undergo secondary fragmentation (Category 4), resulting in finer fractions that are preferentially extracted. The degree of secondary fragmentation is largely influenced by material properties and height of draw (HOD). The larger the HOD and the weaker the material, the more secondary fragmentation is expected and vice versa, illustrated in Figure 16-70. Quantitative forecasting of fragmentation remains largely empirical; even with sophisticated models, the resolution is only sufficient to predict general trends and identify the potential for problems rather than precise outcomes.

Figure 16-70: Reduction of fragment diameter (m) as draw zones grow with extraction

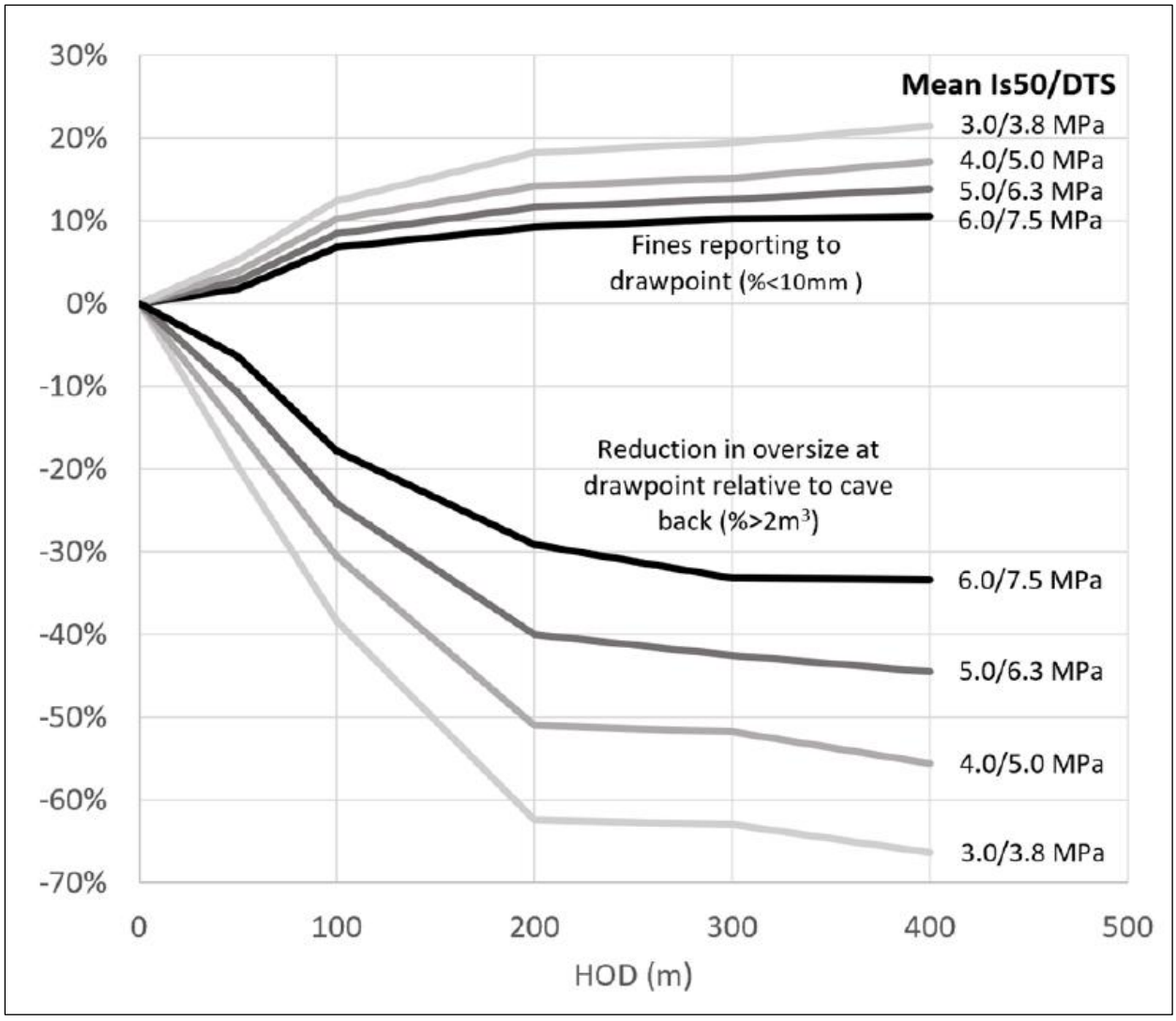


Source: Itasca (2020)

Research by Middindi Consulting (Middindi, 2025) finds most kimberlite has a mean IS50/DTS strength of 6 MPa (see Figure 16-11). Itasca Engineering reports (Itasca, 2020) that this material produces less than 10% fines and under 30% oversize reduction (%<2 m^3) from a 400 m draw height (see, Figure 16-71), based on empirical data. These two datapoints suggest that the cave will have more issues with oversize than undersize. Subsequent numerical model-based estimates (Beck, 2025) indicate a P95 diameter of approximately 1.3-1.5 m for most of the column.

Poor secondary fragmentation may be exacerbated by the fact that all material within the first 170 m HOD will be blasted to relatively fine fragments (90% passing 400 mm) through BAC methods, and that large cave blocks liberated from the cave back may be “cushioned” and pass quickly through this HOD with little secondary fragmentation. In order to mitigate this risk, the project includes mobile secondary breakage equipment, and an automated high-hangup rig for dealing with oversize which bridges higher up in the drawbell (refer Section 16.7.7.2 below).

Figure 16-71: Secondary Fragmentation Estimate



Source: Itasca (2020)

16.7.7 Cave Management

16.7.7.1 Draw Control

The extraction design uses multiple drawpoints for effective muck pile management and dilution control. Continuous but controlled mucking keeps the pile moving, reducing re-compaction and the risk of deadweight above panels. The goal is to achieve an even rate of draw across the whole footprint, to reduce the risk of early dilution entry, and manage air gap. Drawing

consistently from every drawpoint also mixes any pockets of water with dry material, lowering the risk of mud or water rushes.

To mitigate sediment settling and water build-up within the cave, dedicated remuck drifts totalling 280 m will allow for temporary storage of drawn material during planned or unplanned shutdowns of the material handling system. These storage facilities are designed to allow 1 bucket of material drawn from each open drawpoint per 24-hour period for up to 10 days.

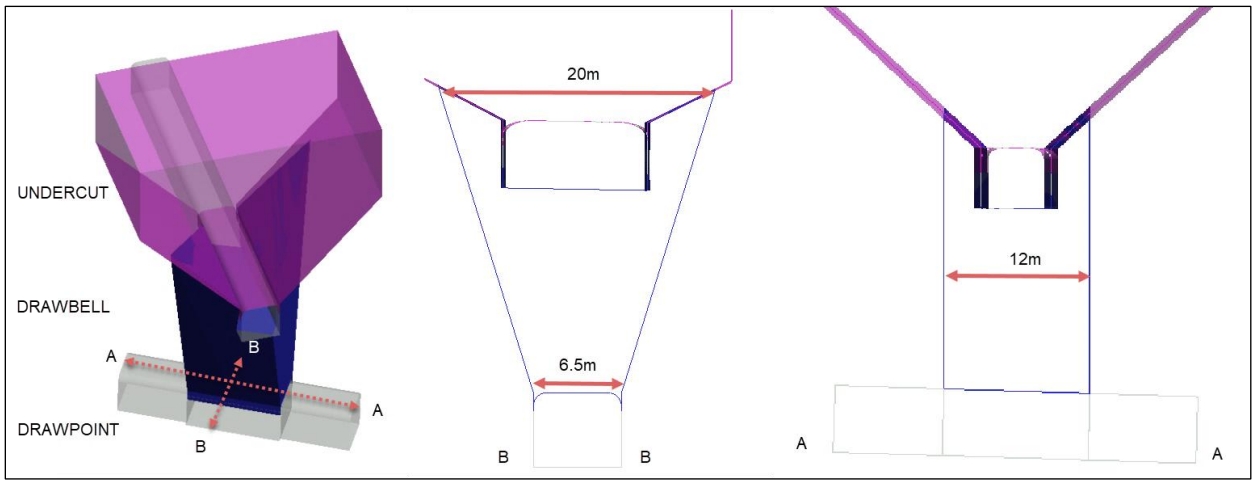
16.7.7.2 Oversize

Oversize material reporting to drawpoints is a common occurrence in cave operations, especially at the onset of caving before secondary fragmentation assists to break down material as discussed in Section 16.7.6.2. Oversize may also result during the BAC blasting phase from blasting malfunctions, excessive blasthole spacing or burden (by design or due to deviation), or material falling from muck-confined free faces or undercut ring toes.

Typically, oversize which requires secondary blasting/breaking intervention at the drawpoints in block cave mines is defined as blocks > 2 m^3. Larger blocks may become jammed higher up in the conical drawbell, requiring either specialised high hang-up automated rigs for drilling and blasting, or drilling from the extraction level.

The maximum width of the drawbell between apex pillars is 20 m x 12 m, with the drawbell narrowing to an opening of 6.5 m x 12 m at the drawpoint elevation, illustrated in Figure 16-72. Blocks reporting to the drawpoint but too large to be picked up by LHD buckets will be broken manually using a mobile rock breaker. Blocks too large to report to the drawpoint will get stuck in the drawbell and will be drilled and blasted with an automated high hangup rig. Drawpoint availability takes into consideration assumptions for hangup frequencies once every 1,000 tonnes drawn, and oversize frequencies once every 500 tonnes drawn from each drawpoint.

Figure 16-72: Vertical Section Showing Maximum Drawbell Width



The greatest risk of oversize comes from fall-off from the muck pile-confined free face of the long-hole blasting rings, and initial free cave material. The former is not likely to be a frequent occurrence, given the general competency of the kimberlite, but in both cases, there is the potential for large blocks to become free and cause a bridge high up in the drawbells, above the drawpoints. In these cases the oversize will be drilled and blasted from the extraction level.

Additional factors have been considered in the design and layout of the mine plan for managing oversize material, including:

- Break-back from the sides of the active caving zone typically generates coarser material, slowing recovery of material outside the initial cave path;
- Given the relatively small number of drawpoints, there is limited space and operational flexibility for managing coarse material on the footprint. The mine design allows for front and rear access to the extraction drives.
- The crusher, tipple, and pass configuration have similar tolerances for coarse material to the existing surface plant, with rock breaking required for all material greater than 1 m diameter. The tipple has been designed with three points of access, a double width grizzly, and dual rock breakers to ensure oversize is managed without bottlenecking the tip. The extraction drive has been designed with ample temporary storage for oversize should mobile breakers be required to assist; and
- Drawbells by their nature are susceptible to significant hang-ups and, as noted above, this has been considered in productivity calculations and the budgeting of mobile breakers and blockholers.

16.7.7.3 Mudrush

A mudrush in a cave mine is a sudden, uncontrolled inflow of water-saturated fine rock material (wet muck) into an underground opening, analogous to a surface debris flow. This phenomenon poses significant risks to mine safety and operations, including injury or death, damage to equipment, production delays, and loss of reserves.

Mudrush is typically caused by an increased water content in the broken material within the cave, and/or an increasing fines content within the cave. Water infiltration may come from groundwater when caving below the phreatic surface, or through precipitation where there are hydraulic connections to surface. Typically, as caves mature and secondary fragmentation increases with height of draw, fines content also increases which can settle and cause water build-up and likewise flow quickly when liberated.

Mudrushes are often triggered by mucking at a drawpoint, blasting or seismic vibrations, or geotechnical failures that cause sudden material release. Management of mudrush is critical to the success and safety of caving operations, and Trigger Action Response Plans (TARPs) are typically enforced to give employees clear and direct instruction for monitoring and responding to changing conditions.

Two TARPs in Table 16-19 and Table 16-20 offer general triggers and actions for cave mining, based on the Argyle Mine In-Rush TARP (Saepulloh, 2022) and the Karowe Cave Management Plan (Beck, 2024). These TARPs will be revised to reflect site-specific conditions as more data

on fragmentation and hydrogeology/hydrology become available. The following TARPs focus on stormwater inflow to the mine and drawpoint muck conditions, however, additional TARPs must be prepared for other potential caving related events such as air blast and drawpoint blockages.

The open pit will be hydraulically connected to the underground workings following breakthrough of the cave, expected in 2029. This hydraulic connection will increase the risk of in-rush (large volumes of water), and potentially inundation of the mine workings during rainfall events. Other mechanisms may also lead to premature hydraulic connections including chimneying along weak contacts, ratholing from uneven draw, unplanned surface subsidence, or caving through saturated host rock or aquifers.

Detection mechanisms that must be employed to monitor and trigger a response include the following:

- A rain gauge placed in the pit;
- A water balance maintained in the underground mine indicating changes to waterflows reporting to the sumps;
- Visual inspection at development faces;
- Visual inspection at drawpoints for wet muck and/or running water;
- Pore pressure monitoring;
- Visually observed changes in draw which might indicate a plug; and
- Visually observed changes in draw indicating rapid material flow, or rapid increases in the amount of fines reporting.

Once an unusual condition has been observed, the appropriate trigger level will be determined and the corresponding actions will be put in place including notification of all responsible parties.

In addition to the TARP actions, it is important to consider information on flow disruptions from drawpoints, which may signal hang-ups caused by compaction and possible mud plugs. These situations are especially vulnerable to increased water inflow, as increased head can trigger mudrushes of varying sizes. Monitoring piezometric pressures within the country rocks is also essential to detect any sudden rises in pore pressure, indicating more water entering the muck-pile.

Table 16-19: Stormwater TARP

Level/Condition	Trigger	Risk	Action	Responsible	Reporting
Dry Season	**<200 mm Cumulative Rainfall**	No In-rush/Mud-Rush Risk	• No Action Required	• Operator • Shift Boss	• No reporting
Wet Season	**Cumulative Rainfall >=200 mm**	In-Rush/Mud-Rush Warning – TARP Activated	• Clearance from Shift Boss prior to mucking • Production Superintendent, Senior Geotech, and Hydrogeologist to review the draw percentage • Operators to communicate every 2 hours to Shift Boss on draw point condition (fines content and dampness) with any changes to be reported immediately • Conventional mucking to be used	• Operator • Shift Boss • Geotech Team • Hydrogeologist General Manager	• Internal Reporting • Notify Ministry of Mines • Notify Senior Management
Wet Season Cumulative Rainfall >=200 mm	**Rainfall >= 25 mm in 1 hour Or Rainfall >= 40 mm in 2 hours Or Rainfall >= 50 mm in 3 hours**	Potential In-Rush/Mud-Rush Hazard	• All personnel to be evacuated to surface and tag board cleared • A minimum exclusion time of 8-hour is required • A minimum of 8% of water from the rain event shall have been reported to the U/G inflows as measured by the mines water balance	• Operator • Shift Boss • Geotech Team • Hydrogeologist • General Manager • Senior Management Team	• Internal Reporting • Notify Ministry of Mines • Notify Senior Management

Table 16-20: Drawpoint TARP

Water Content		
Dry Draw Point (1)	Damp Draw Point (2)	Wet Draw Point (3)
The whole rill surface is dry and no water flowing from the draw point.	Rill surface is <= 100% damp, but no water flows from the draw point	Rill surface is 100% damp; water flow is observed flowing out of the draw point.

Risk Ranking			
Water classification at the draw point	Percentage of fines material less than 5 cm in diameter (P50)		
	Less than 30% of the material is fines (mostly coarse material greater than 50 cm)	Between 30% and 70% of the material is fines (mixtures of fines and coarse, 5 cm to 50 cm size)	More than 70% of the material is fines (mostly fine material, less than 5 cm size)
Dry (1)	A1	B1	C1
Damp (2)	A2	B2	C2
Wet (3)	A3	B3	C3

Trigger Condition	Action	Responsible	Reporting
Green	• No clearance required from Shift Boss prior to mucking • Shift Boss must check the draw point status after each ring firing • Conventional mucking is to be used • No specific signage or additional supervision is required	• Operator • Shift Boss	• No reporting
Yellow	• Clearance required from Shift Boss prior to mucking • Superintendent and Senior Geotech to review draw percentage • Operators to communicate every 2hrs to Shift Boss on draw point condition and report changes immediately • Tele-Remote Mucking only	• Operator • Shift Boss • Geotech Team • Hydrogeologist • General Manager	• Internal Reporting • Notify Ministry of Mines • Notify Senior Management
Red	• No unauthorised entry to ore drive • Superintendent and Senior Geotech to review draw percentage • Draw point condition must be checked by Shift Boss and Geotechnical Engineer • Draw point condition (water inflow and fines content) to be re-evaluated by loader operator on an ongoing basis and by Shift Boss every two hours • Risk assessment before mucking to determine appropriate hazard controls (evacuation potential) • Tele-Remote Mucking only • Production engineer to update master file and production layers for hazard risks following notification by Geotechnical Engineer/Hydrologist	• Operator • Shift Boss • Geotech Team • Hydrogeologist • General Manager • Senior Management Team	• Internal Reporting • Notify Ministry of Mines • Notify Senior Management

16.7.7.4 Subsidence

Cave simulations indicate low subsidence potential outside the OP perimeter. Early 2025 simulations (Beck, 2025) showed ground disturbance west of the pit several years after production has completed. This disturbance encroaches on the current 11 kV power supply to the shafts. Considerations should be made for potential subsidence before placing any further permanent infrastructure near the OP perimeter. Simulations show no risk to areas east of the OP, however, stockpile drawdown should prioritize those located closest to the pit perimeter first to eliminate risk potential as the cave propagates. Monitoring equipment as discussed in Section 16.3.14 will provide early warning systems for ground disturbance.

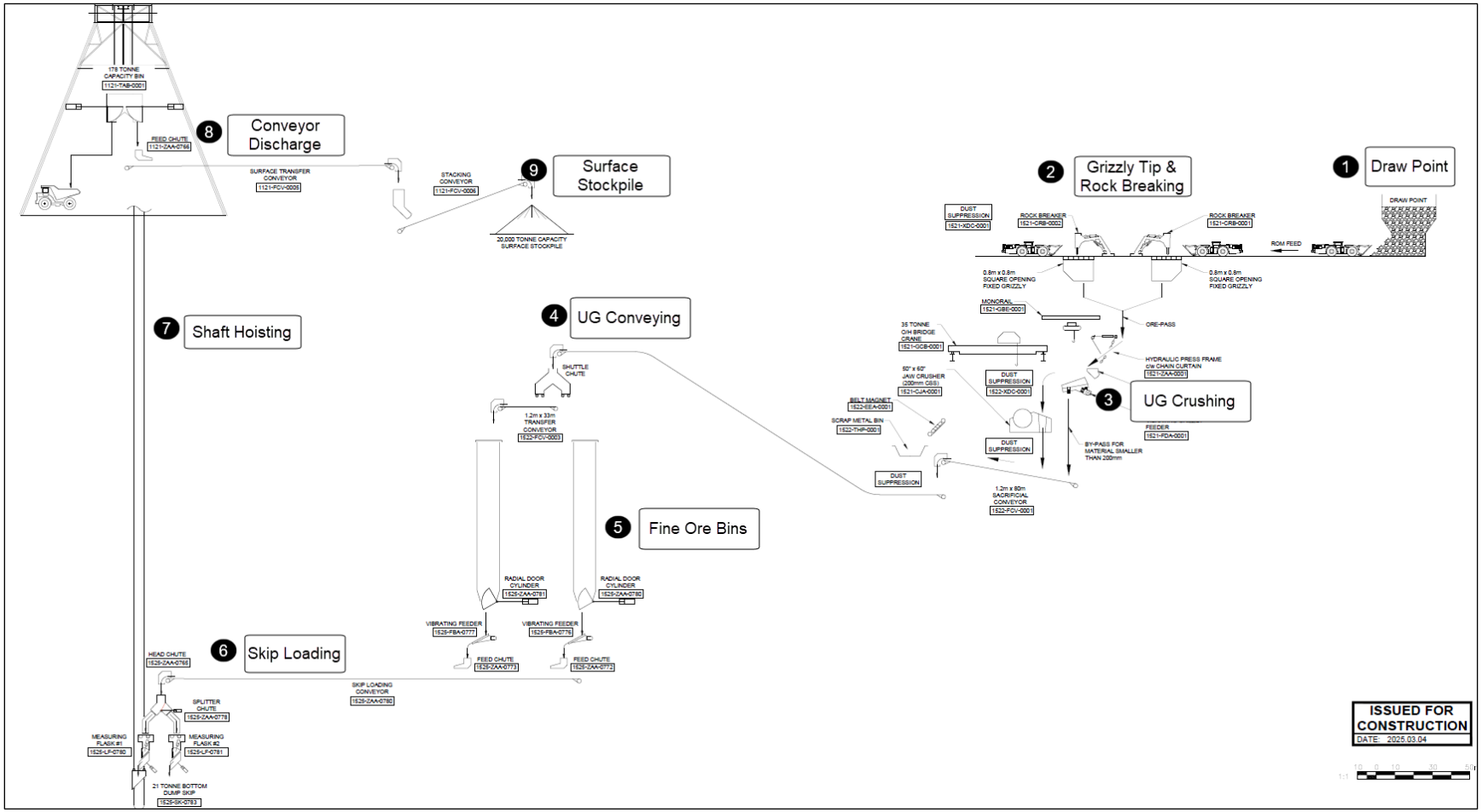
16.8 Mine Services

16.8.1 Materials Handling

The materials handling system consists of single stage crushing and UG conveying to a double drum skip hoisting system.

Figure 16-73 illustrates the UG production material flow from drawpoints to the surface in permanent condition.

Figure 16-73: Underground Materials Handling Process Flow Diagram



Production targets are for 465 t/h or 7,800 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 are outlined below in Table 16-21.

Table 16-21: Materials Handling 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 (2024)

Production LHDs will muck ore directly from drawpoints to a central, double wide, three-sided UG grizzly. Material will be dumped onto a 9.5 m x 5.5 m static grizzly with 800 mm aperture above an ore pass. Oversized material from the static grizzly will be reduced in size by one of two BTI MRH T 20/25 BXR50 tele-remote-ready rock breakers. During initial ramp up an operator will control the rock breaker from a local control booth; but tele-remote operations are envisioned during operations. Difficult to break material will be removed via the LHD or rock breaker and taken to a remuck bay 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 minimise 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. 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 75 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 260 m long, 1200 mm wide loading feed conveyor with an installed power

of 160 kW. A 30 m long, 1200 mm wide transfer conveyor will discharge material, transferred from the loading feed conveyor, into the top of one of two 50 m tall fine ore bins.

The fine ore bins will discharge onto a skip loading level conveyor via hydraulically operated radial gates, vibrating feeders with variable speed drives (VSDs), and discharge chutes. Only one silo is used at a time, selected by the control room operator, with interlocks to prevent complete emptying. The loading level conveyor, which runs at two speeds depending on the feed source, transports ore to a hydraulically operated traverser chute that directs it to one of two measuring flasks. These flasks use dual-bridge load cells to weigh ore batches for optimal skip loading. A side loading arrangement with a bin, grizzly, and rock breaker allows LHD tipping onto the skip loading level conveyor manually when silos are inactive, managed by traffic lights for operator safety.

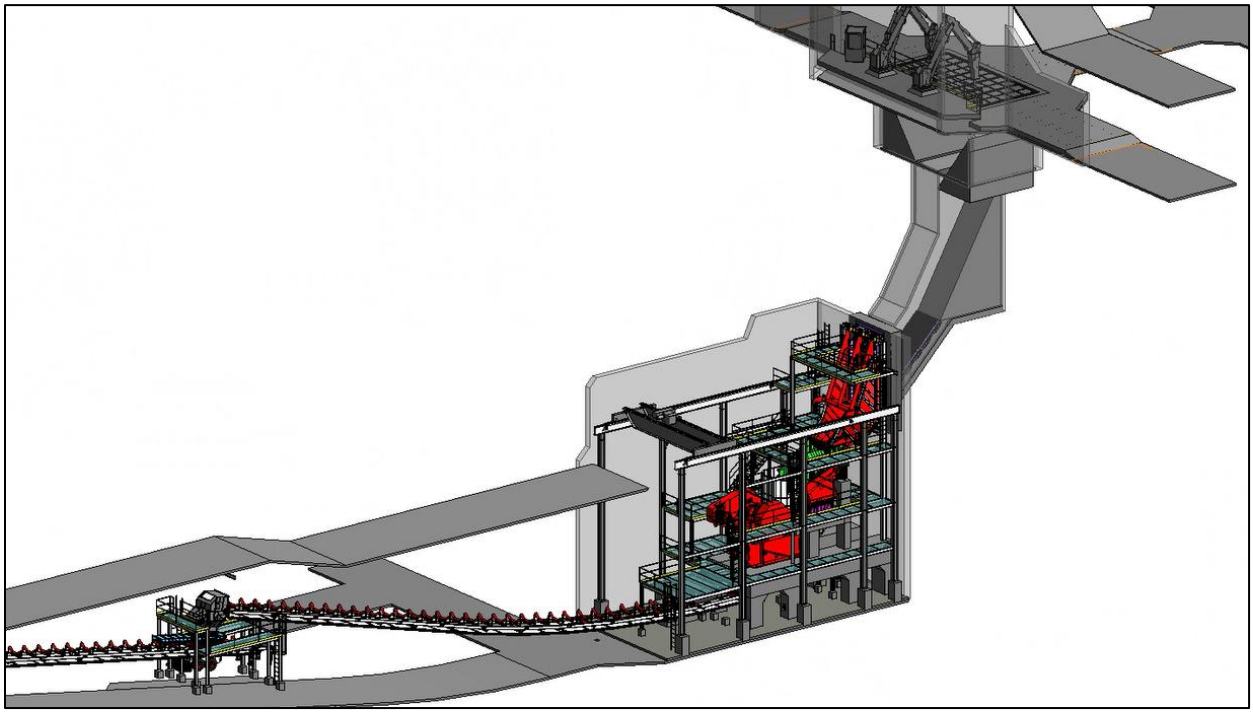
The production winder, controlled via a separate PLC, cycles two (2) 21 tonne skips to transport ore to the surface headgear bin, which temporarily stores ore and regulates feed. The bin has two discharge points: one for truck loading via radial gates and chutes, and another for feeding a discharge chute onto a conveyor to a stockpile. 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.

Dust extraction systems will be introduced at multiple locations in the comminution circuit to capture airborne dust at the source.

The layout of the crusher chamber is outlined below in Figure 16-74.

Figure 16-74: Crusher Chamber Layout



Source: JDS (2024)

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 utilised for water management requirements to incorporate some contingency into the design.

Once a connection is made between the OP and the UG workings, expected in Q3 2029, the UG workings will also need to manage inflows from precipitation. The OP will provide a catchment area for any precipitation, in addition to any groundwater entering the OP. These volumes of water will have the opportunity to flow into the UG mine from the OP and to travel down via

fractures or lithological contacts (or, once the cave breaks through, via the cave) to the open cavities of the UG mine.

Precipitation is estimated into normal rainfall and stormwater volumes. Rainfall volumes are generally low in the region. Stormwater volumes, however, can be considerable over 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 16-22.

Table 16-22: 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	69,000

Based on the inflows estimated from the three primary sources, the estimated peak inflows from each elevation are outlined below in Table 16-23. 2030 represents the year of the highest daily inflows, after which estimated inflows begin to decline over the remaining LOM.

Table 16-23: Peak Daily Groundwater Inflows by Level (m^3/day) up to 2030

Date	Peak (each)	Avg (each)	2025	2026	2027	2028	2029	2030	2031	2032	2033
Groundwater Inflows											
670 Shaft Water Ring	144	144	144	144	144	144	144	144	144	144	144
285 Shaft Water Ring	144	144	144	144	144	144	144	144	144	144	144
670	494	117	39	31	31	494	480	306	226	43	28

Date	Peak (each)	Avg (each)	2025	2026	2027	2028	2029	2030	2031	2032	2033
470	1,183	543	72	240	1,172	1,183	986	760	717	575	475
405	277	70	-	33	277	173	97	63	56	49	44
340	220	43	-	32	220	54	38	33	30	28	26
310	6,544	4,472	250	239	786	841	2,819	3,916	6,276	6,364	6,477
285	398	163	72	247	398	347	155	139	132	128	125
245	244	146	1	10	15	244	229	198	186	178	173
Mine Service Water											
670	57	42	-	-	26	57	-	-	-	-	-
470	185	129	-	28	185	174	-	-	-	-	-
405	42	36	-	23	42	42	-	-	-	-	-
340	52	34	-	8	52	41	-	-	-	-	-
310	85	28	20	58	85	35	20	20	20	19	20
285	51	34	33	51	24	30	-	-	-	-	-
245	60	28	9	21	24	60	-	-	-	-	-
Infrastructure and Contingency	1,182	1,144	-	-	1,067	1,182	1,182	-	-	-	-
Precipitation											
Precipitation	1,069	1,036	-	-	-	-	706	1,069	1,069	1,069	1,069
Storm Water (1:100)	5,431	5,431	-	-	-	-	5,431	5,431	5,431	5,431	5,431
Total	**13,071**	**9,862**	**652**	**1,198**	**4,080**	**4,313**	**10,061**	**11,001**	**13,053**	**13,071**	**13,008**

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.

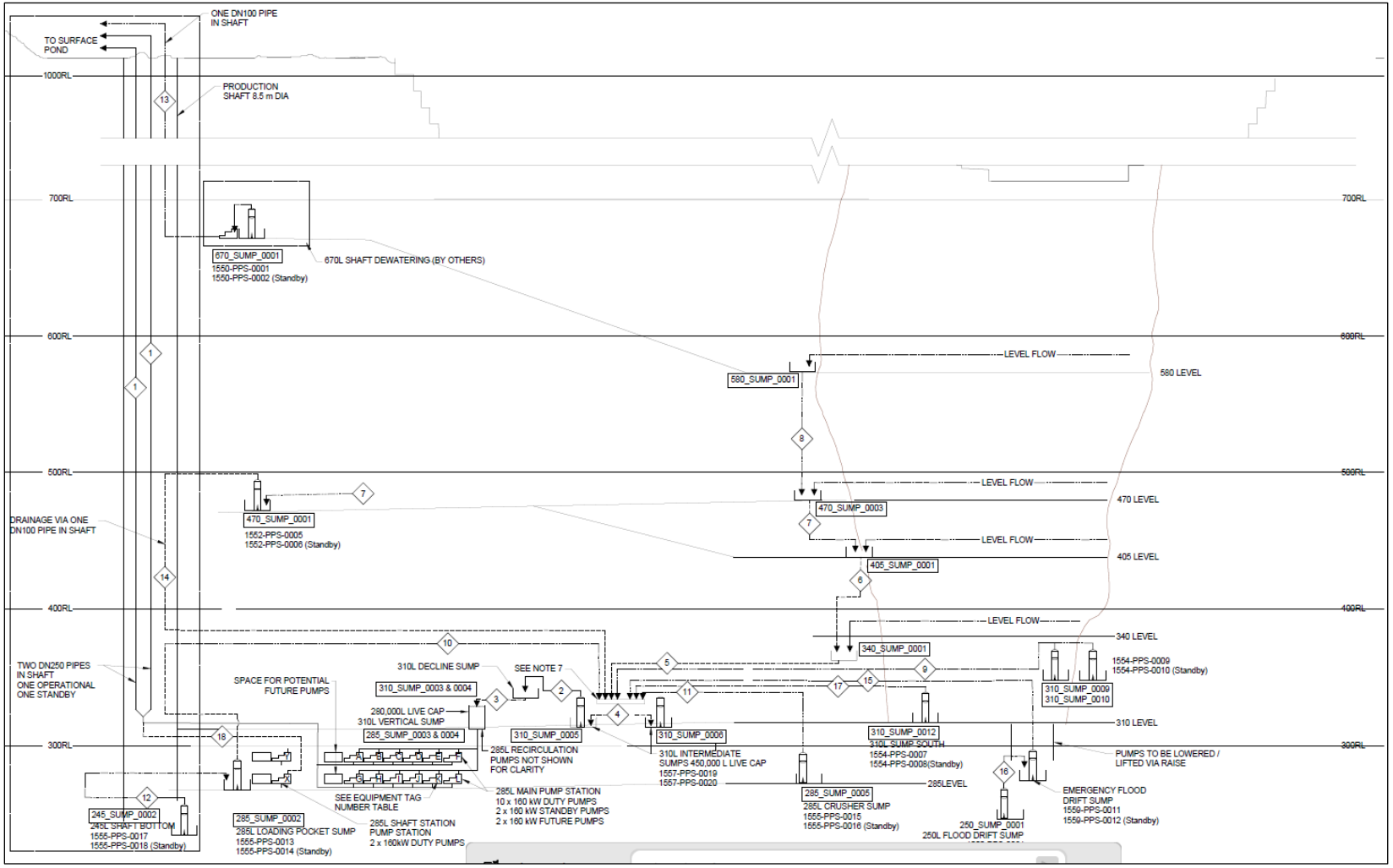
Key specifications for the Dewatering System are outlined below in Table 16-24.

Table 16-24: Key Dewatering System Specifications

Item	Units	Value
General Conditions		
Fluid Type	-	Dirty Water
System Flow Rate (Design)	m^3/hr	550
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

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 16-75.

Figure 16-75: Dewatering Process Flow Diagram



The Dewatering System cascades and directs the majority of water to a UG Main Pumping Station which consist of eight (8) components:

1. 670 lvl Pump Station;
2. 285 lvl Shaft Station Pump Station
3. Level Sumps;
4. 310 lvl Intermediate Sump;
5. Vertical Sump;
6. 285 lvl Main Pump Room;
7. 285 lvl Stormwater Pump Room, and
8. 250 lvl Flood Drive Storage.

16.8.2.3 670 lvl Pump Station

During shaft sinking and station development at the 670 lvl, two Scamont FXG 37 kW pumps were installed. 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 utilised UG for service water, and can be utilised at surface without treatment. All dewatering systems below this pump station will be managing highly saline water.

16.8.2.4 285L Shaft Station Pump Station

During shaft sinking and station development at the 285L, two Scamont SP200 160 kW positive displacement pumps will be installed. These pumps will operate in a duty / standby configuration to allow for maintenance. The selected pumps have had the wet end upgraded to duplex stainless steel to prevent against corrosion. These pumps will deliver dirty water to surface in a single lift. Water inflows from lateral development on the 285L and 310L will be redirected to a feed dam supporting the 285L shaft station pump station, until the permanent dewatering system can be constructed.

16.8.2.5 Level Sumps

A series of typical decline and borehole sumps will be used throughout the mine to cascade water to the 310 lvl 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.

16.8.2.6 310 lvl Intermediate Sump

All mine water will report to a pipe manifold at the 310 lvl 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 lvl 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.

16.8.2.7 Vertical Sump

Two vertical sumps shall be located between the 310 lvl intermediate sump and the main pump room. The vertical sumps will be a 4 m x 4 m excavation spanning from the 310 lvl down to the 285 lvl with a vertical hydrostatic bulkhead located at the bottom, separating the sumps from the main pump station on 285 lvl. 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.

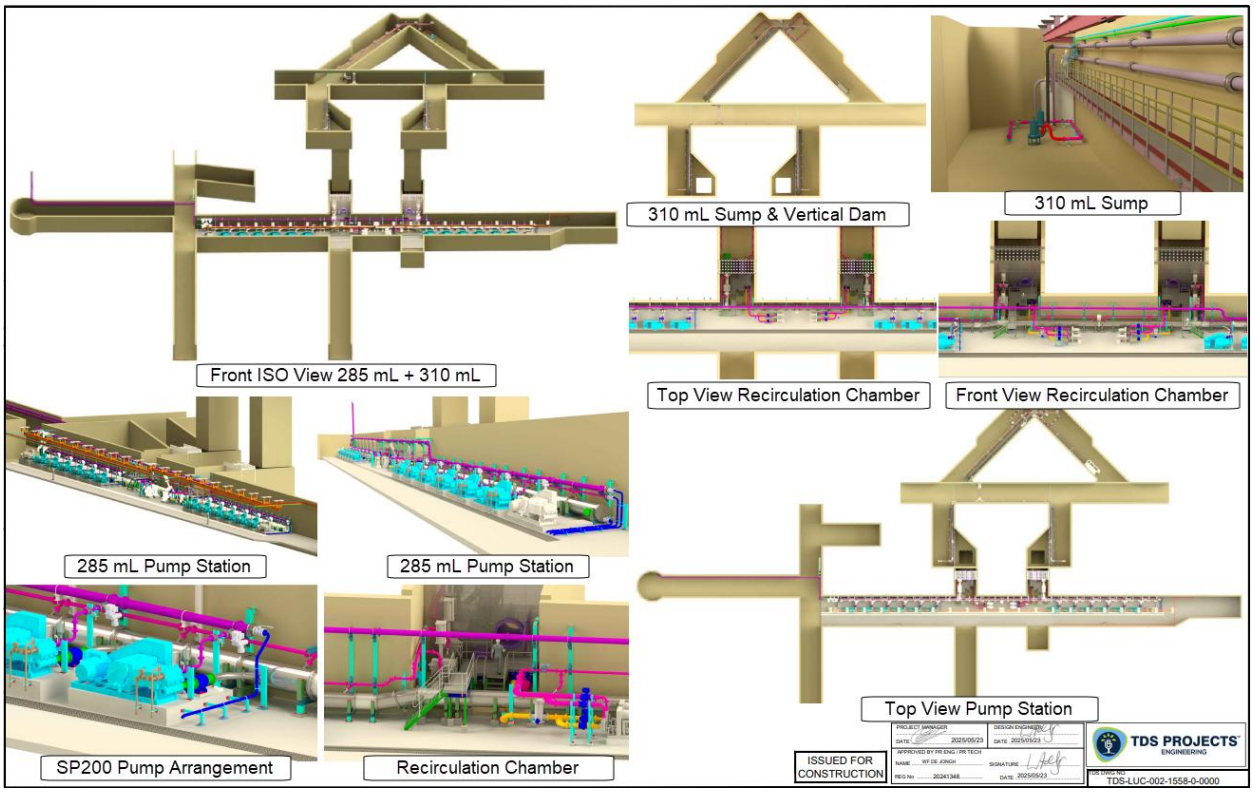
16.8.2.8 285 lvl Main Pump Room 1

Scamont SP200 160 kW positive displacement pumps have been selected as the main pumps and are to be installed at the 285 lvl Main Pump Room 1. 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 into two pump rooms in order to limit the length of the suction line and allow for a phased construction approach. In Pump Room 1, 7 pumps are to be installed in parallel, with a 5 duty and 2 standby pump configuration.

The 285 lvl Main Pump Station will deliver water to surface through either a 100 NB or 250 NB pipe in the P/S depending on the rate of water inflow. A second 250 NB pipe will be installed as a standby. The surface water management facilities are outlined in Section 18.2.

A dedicated electrical room will be required for the Main Pump Station and will be positioned nearby the 285 lvl Main Pump Room 1. The control of the main pumps will be automated, however, the mine intends to operate with full time pump operators, monitoring and maintain the main pumps frequently. The general arrangement of the 285L Main Pump Room is outlined below in Figure 16-76, and can be seen on the level plan under Figure 16-58.

Figure 16-76: 285L Main Pump Room #1 General Arrangement



Source: TDS (2025)

16.8.2.9 285 lvl Main Pump Room 2

To manage potential stormwater inflows in late 2029, an expansion to the 285 lvl Main Pump Room is planned. A second Main Pump Room will be constructed in a similar configuration as Main Pump Room #1. An additional 7 Scamont SP200 160 kW Positive Displacement pumps will be installed in parallel, fed from the common suction header connected to the vertical sumps. The dedicated electrical room for Main Pump Room #1 will be expanded to incorporate the equipment required for this expansion.

Due to the potential for large stormwater inflows, a watertight door at the 285L has been incorporated into the design, identified in Figure 16-60. This will provide approximately 59,600 m^3 of flood storage capacity, reducing the required flow rate for the expansion of the 285L Main Pump Room. After a 1:100 storm event, it is estimated that the excess dewatering system capacity will be able to dewater the flood storage in two weeks.

16.8.2.10 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 lvl using HDPE piping. At the high temperatures expected from the groundwater inflows (see Section 16.8.4), the HDPE pressure rating will be derated significantly. In the 285 lvl 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.

16.8.2.11 Water Disposal

From the UG operations water will be pumped to a surface water management facility, which is outlined in Section 18.2.

16.8.3 Mine Ventilation

16.8.3.1 Introduction

The ventilation system for KDM UGP is based on a planned production rate of 2.85 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 the sole exhaust route. 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. Figure 16-77 depicts the primary ventilation network of the mine during steady-state production (year 2029 through to end of LOM).

The mine ventilation will operate as a pull system with one fresh air source and one return air route. 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.

Peak airflow demand for the UGP is 422 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 bifurcated 800 kW fans have a combined airflow capacity of 450 m^3/s (at a pressure of 3.0 kilopascal (kPa)), 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 the 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 422 m^3/s is based on a build-up of the individual active zones, stopes, and other areas (bottom-up approach), and on the diesel equipment fleet requirements (top-down approach). 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 (Howden, 2022).

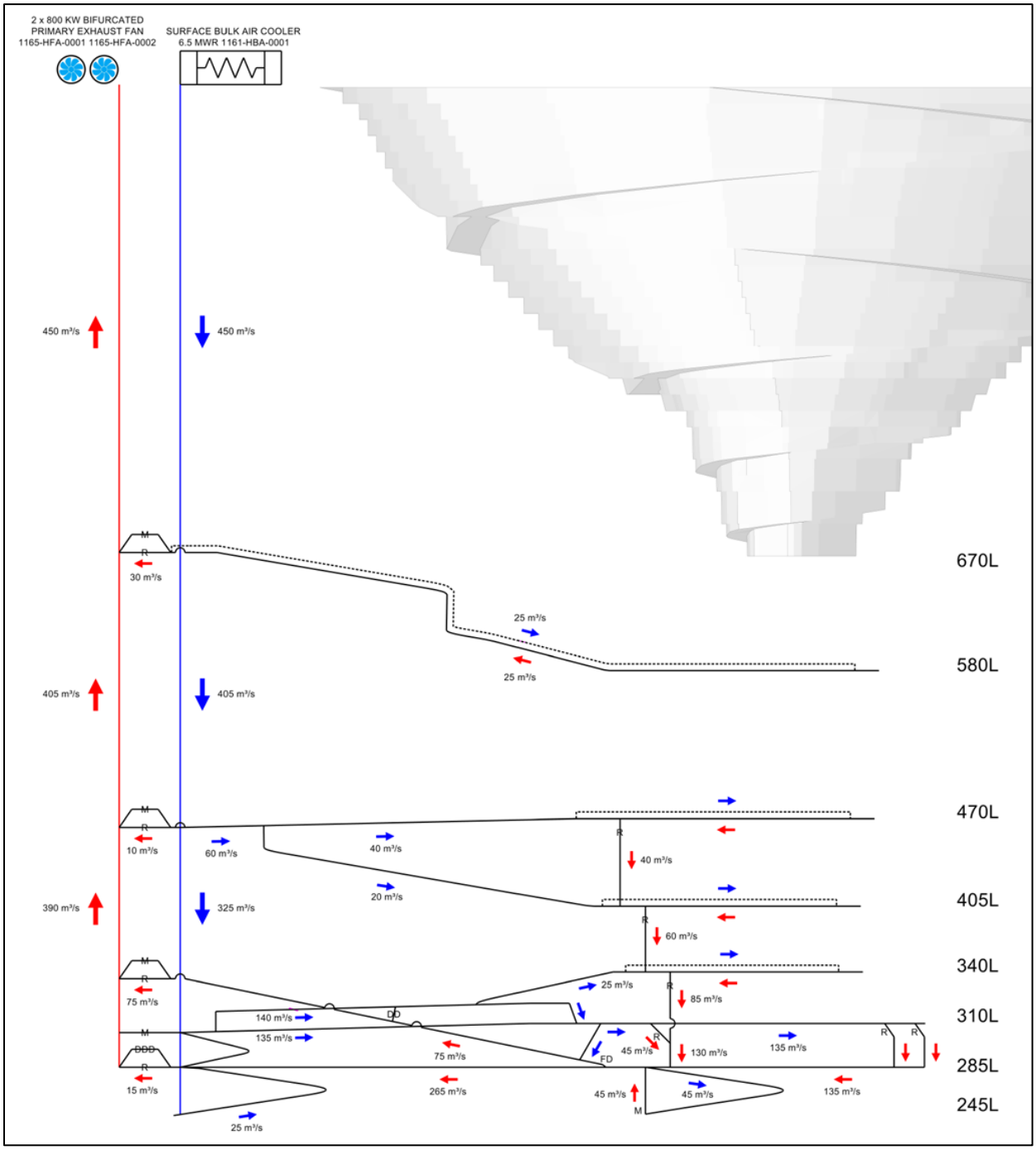
Primary ventilation support is achieved using bulkheads, conventional airflow regulators, and airlocks to allow personnel and equipment to move from circuit to circuit without disrupting the overall ventilation system. Ventilation distribution management is based on using pressure drop regulators (passive 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) system 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 in under five (5) years of operation, however, to keep mine ventilation systems simple and cost-effective, no VOD systems have been incorporated into 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 three water-cooled chillers and two air-cooled chillers.

Figure 16-77: KDM UGP Ventilation Long-Section



16.8.3.2 Ventilation Design Criteria

The main design criteria used to configure the UGP ventilation system include the bases of diesel-powered equipment ventilation, velocity requirements, airflow allocation for underground infrastructure including fixed facilities, sizes and k-factors for different airways, auxiliary ventilation criteria, and others.

The basic minimum ventilation rate for the dilution of diesel exhaust gases and particulates per Botswanan MQMA regulations 108-f is 0.05 m^3/s fresh air per kW of all diesel units in operation. It should also be noted that power-referenced specific ventilation rates are a single factor and other criteria, such as minimum velocity, may drive the actual ventilation rate.

Airflow velocity limits are listed in Table 16-25 below.

Table 16-25: Air Velocities

Design Item	Recommended Criteria
Conveyor Drift (Homotropal)	0.75 m/s – minimum (relative to the conveyor) 5.00 m/s – maximum
Ramp / Truck Haulage Drift	0.75 m/s – minimum 6.10 m/s – maximum
Fresh Air Drift (with personnel)	6.10 m/s – maximum 7.10 m/s – Absolute maximum
Return Air Drift (dedicated and without personnel)	10 m/s
Production Shaft	12.7 m/s – maximum – with rigid guides
Ventilation Shaft / Return Air Raises (RAR) (upcast)	20 m/s –maximum If water/condensate is a factor, avoid 6.8 - 12.2 m/s to prevent water blanket formation
Fresh Air Raises (FAR) (downcast)	20.0 m/s – maximum – No people 12.7 m/s – maximum – with people
Emergency Egress	0.5 m/s – minimum 4.0 m/s – maximum
Production Areas	0.75 m/s – minimum 2.00 m/s – maximum
Crusher	0.5 m/s – minimum
Rigid Duct	20.3 m/s – maximum
Flexible Duct	11.4 m/s – maximum

The K-factors and resistances for different airways and the ventilation controls in the mine are listed below in Table 16-26.

Table 16-26: Airway k-Factors at Standard Air Density of 1.2 kg/m^3

Design Item	Recommended Criteria
Production Shaft	0.0075 (Concrete lined shaft equipped with steel sets)
Ventilation Shaft	0.0050 (Concrete lined shaft)
Bored Raise	0.0050
Slashed Raise	0.01 (Auto calculated in VentSim^{TM})
Blasted Drift	0.0150 (Auto calculated in VentSim^{TM})
Vent duct, flexible round	0.0029
Vent duct, Rigid Plastic	0.0013
Vent duct, Spiral Steel	0.0037

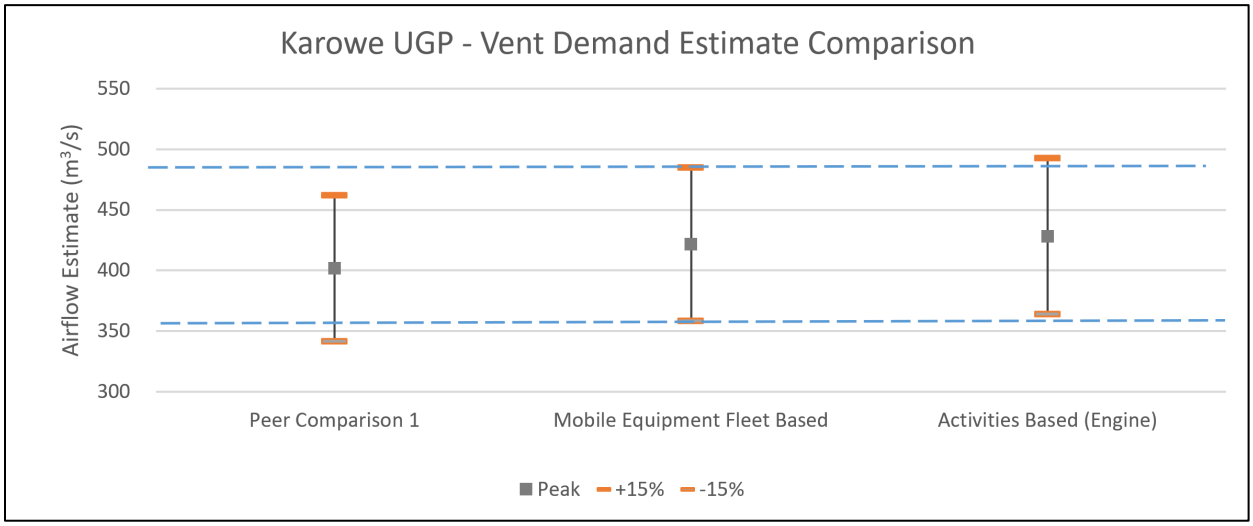
16.8.3.3 Ventilation Demand Estimate

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 (QP internal database);
- Approach 2: First principles – based on diesel equipment fleet (Top-down Method); and
- Approach 3: First principles – based on workplace activities / crews (Bottom-up Method).

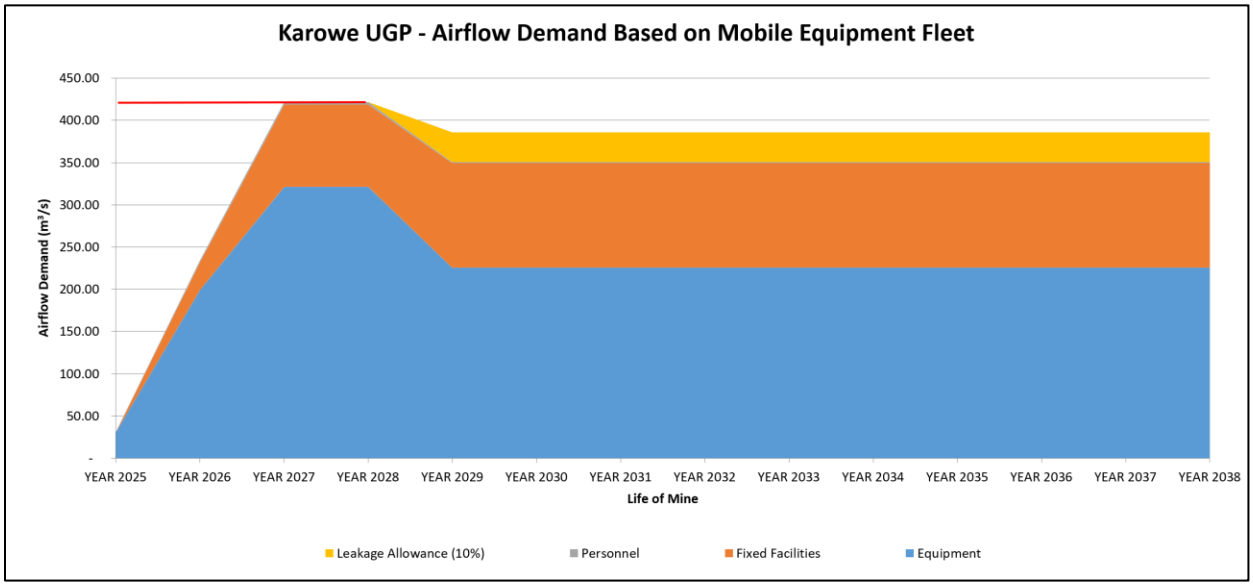
The final estimate is based on Approach 2 (Diesel Fleet based), which indicates a peak requirement of 422 m^3/s. The estimated flow based of these approaches is summarised in the graph below (Figure 16-78). The surface main fans are sized based on the peak ventilation demand.

Figure 16-78: Ventilation Demand Estimate Comparison



The ventilation demand profile based on diesel equipment fleet over the LOM is shown in Figure 16-79. 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. The airflow requirement estimated in this approach is taken as the basis for designing the ventilation system and sizing the main fans.

Figure 16-79: LOM Ventilation Demand Profile – Equipment Fleet Based



16.8.3.3.1 Main Fans

VentSim™ modelling was performed at various development and production stages to determine the sizes of the main fans. During steady state production phase, a total of 415 m^3/s enters the mine through the P/S. After accounting for auto compression and system leakage, the resulting UG airflow achieves the target demand of 386 m^3/s.

The main fans will be equipped with variable frequency drive (VFDs) to be able to operate at various operating points over the LOM as necessary. The fans will also have temperature and vibration monitoring equipment, and PLCs to control the airflow from a local HMI.

Fan motors will be externally mounted and connected by drive shaft to minimise corrosion through interaction with the saline water expected UG. Exhaust discharge will be vertical to minimise noise on the working terrace. Exhaust fans will be situated southeast of the fresh air intake, against the prevailing wind direction, to minimise 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). Baffles will be provided in front of the main fans to allow ejection of air blasts arising from large production blasts without damaging ventilation equipment.

One 75 kW emergency fan will be installed in a bulkhead immediately in front of the V/S on 670 lvl, 470 lvl, and 310 lvl shaft stations to facilitate emergency evacuation and to provide backup ventilation in the event of failure of the main fans.

16.8.3.4 Auxiliary Ventilation

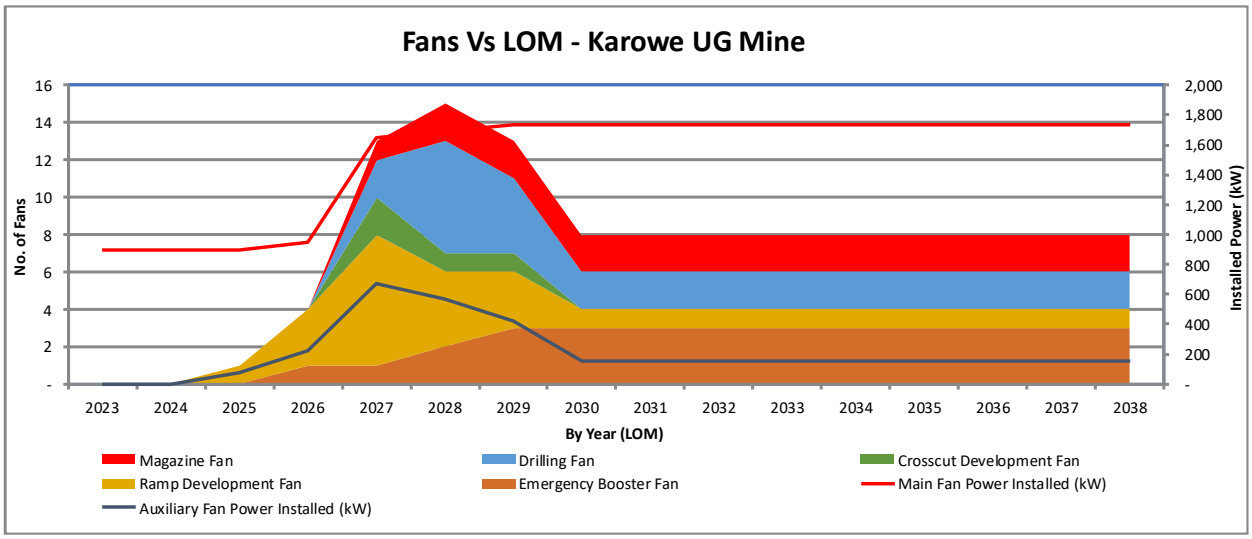
UG mining activities and fixed facilities will require auxiliary ventilation. The airflow requirement for each fan is based on the crew that fan needs to support, the size of the duct, length of the duct through which the airflow is pushed, and the duct leakage. The duct is sized to accommodate the duct hung in the back 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 21 auxiliary fans are required with power ratings between 4 kW to 75 kW to meet the needs of the mine plan. As mine areas are decommissioned, ventilation infrastructure will be relocated to new areas.

16.8.3.5 Total Ventilation Power Demand

The total peak installed ventilation power demand is estimated at 2.32 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 (years 2027→2029). The respective peak electrical demand to operate the fans is approximately 1.79 MW.

LOM ventilation power is illustrated in Figure 16-80. Assumptions include electrical demand for the main and booster fans at 80% of the installed power, and 70% for the auxiliary fans.

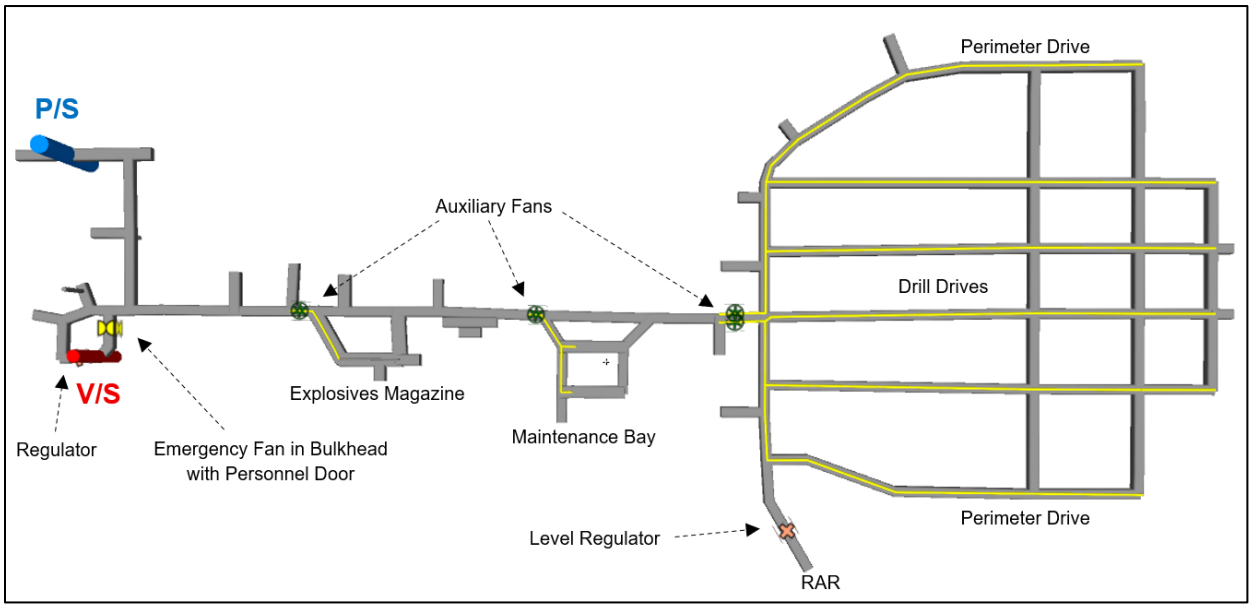
Figure 16-80: Fans and Electrical Load Profile Over LOM



16.8.3.6 Primary Ventilation Controls, Regulators, Doors, and Bulkheads

Each level of the KDM UGP has multiple ventilation controls and monitoring equipment. Figure 16-81 shows the arrangement of these infrastructure on a typical level at the KDM UGP.

Figure 16-81: Typical Level Ventilation Controls



The following are the various ventilation controls present in the KDM UGP ventilation system:

- Personnel Door: There are 30 personnel doors planned in the system over the LOM. Some are standard personnel doors in bulkheads, while some others are part of regulator and control door installations;

- Control Door and Airlock: There are two electrically operated and sensor-controlled 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;

- Regulators: A total of 16 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 lvl. A combination of louvered regulators and roll-up door regulators are planned. The louvers and roll-up doors can be positioned to a desired setting to maintain a fixed airflow rate or to vary the airflow rate based on need. A personnel door will be located as required in regulator bulkheads 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 processes, development crews carry out lateral development off 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.

16.8.3.7 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.

16.8.3.8 Emergency Alert System

In the KDM UGP, a “stench gas” system will be provided to give an emergency 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 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 characterised 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, with groundwater temperature recordings exceeding 45°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.

To ensure workplace temperatures are kept below 27.5°C Wet Bulb Globe Temperature (WBGT) and/or level reject temperatures below 28.5°C wet bulb, a 4.5 MWR Bulk Air cooling plant was constructed in 2024, as shown in Figure 16-82.

The plant comprises of a bulk air cooler (BAC) with three (3) 1.5 MWR water chillers and expansion capacity to 9.0 MWR with future addition of three (3) electric air chillers, two of which are currently on site and servicing the shafts in a temporary sinking arrangement.

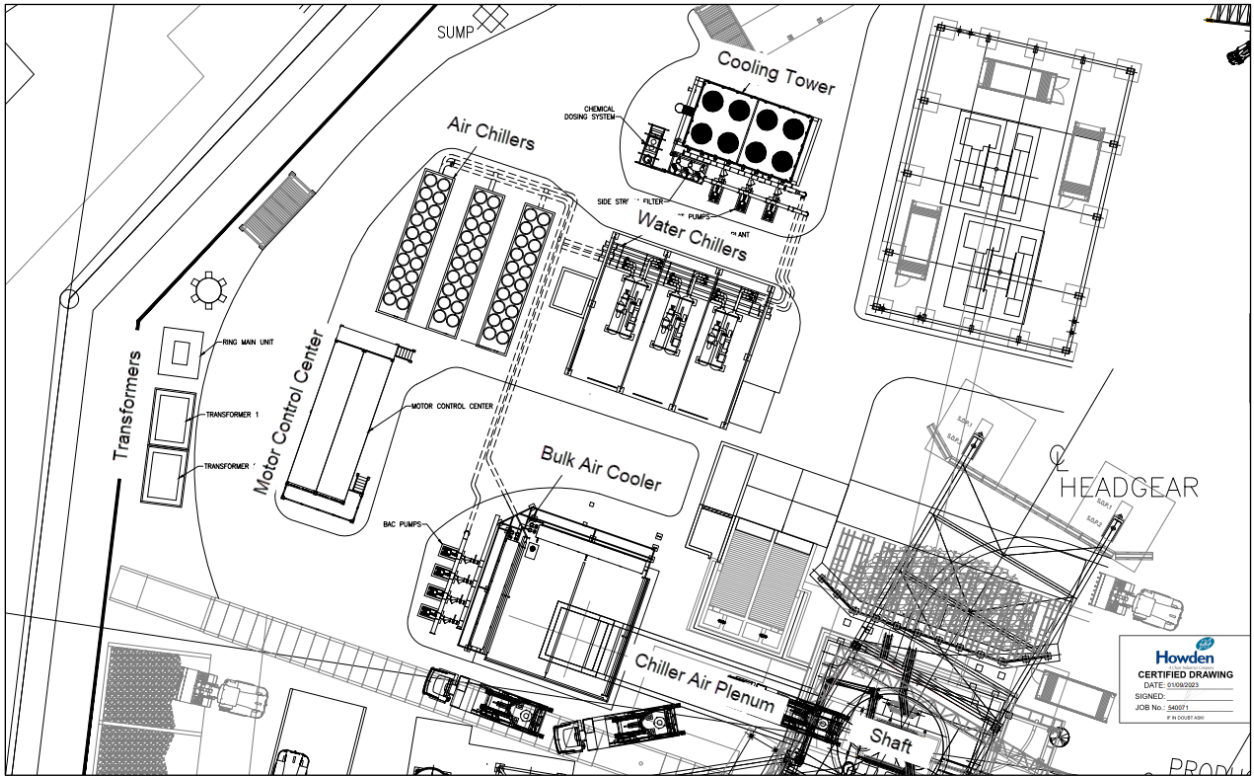
A subterranean concrete plenum of 4 m x 4 m dimension will serve as chilled air intake to the P/S. On top of this plenum sits a large diameter forced air ventilation fan capable of pushing 163 m^3/s (178 kg/s) into the P/S barrel. Surrounding the plenum fan a fully enclosed bulk air cooler with large cooling coils mounted to the walls chills fresh air as it is drawn into the plenum.

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 water 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 will be supplied by the existing mine water treatment facility, which 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 450 m^3/s. The BAC plenum fan will push 163 m^3/s of chilled air into the shaft barrel and the remaining 287 m^3/s will be drawn from the Production Shaft collar at an ambient temperature of 29°C, mixing to provide an average intake temperature of 22°C.

Figure 16-82: Refrigeration Plant at P/S Collar



Source: Howden (2023)

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 16-27).

The peak refrigeration demand is estimated based on the worst case (design basis) surface temperature and thermal load from the mine.

Table 16-27: 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)	1. 140 W/m^2 unrestricted work 2. 140 - 115 W/m^2 workplace precautions required 3. < 115 W/m^2 shift lengths need to be monitored

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 onwards) is 2.3 MW.

16.8.4.2.2 Electrical Equipment

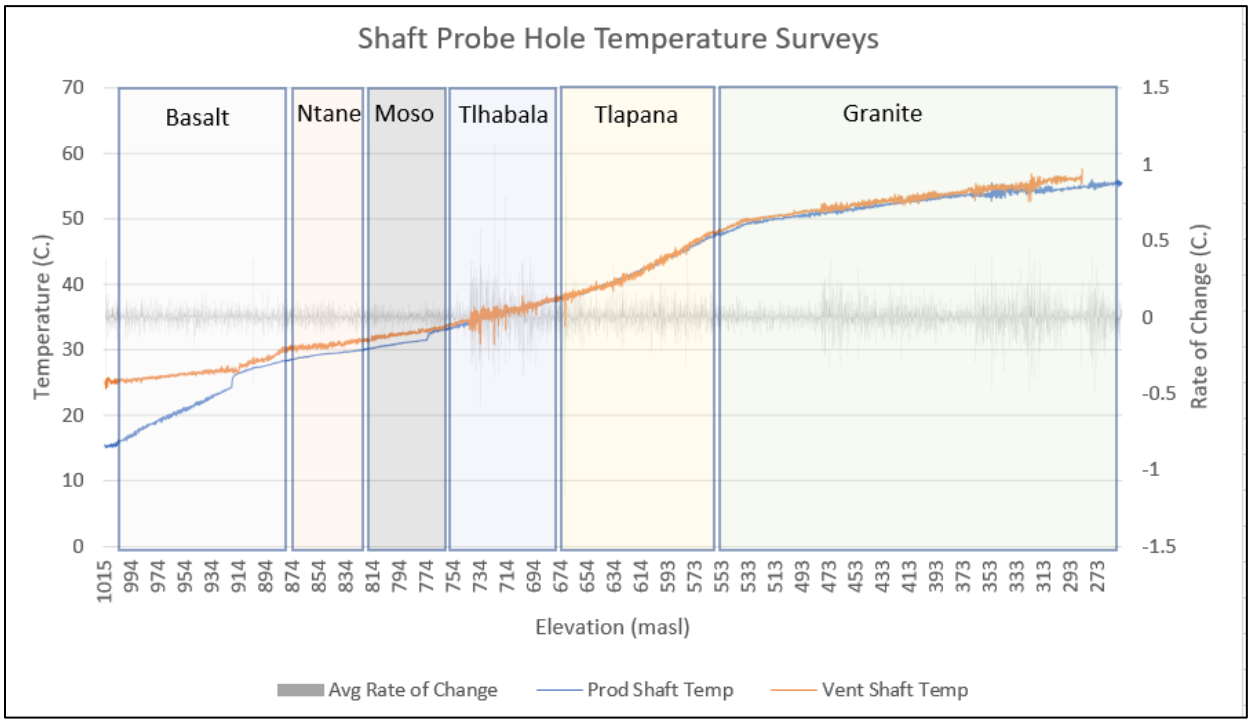
For electrically powered equipment, such as auxiliary fans, all electrical energy consumed is rejected as heat. This holds true for other electrically powered equipment as well, except for pumps when a portion of the consumed energy is used to move water. The total estimated heat load is 1.1 MW.

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 16-83.

Figure 16-83: Geothermal Gradient



Source: Poseidon (2021)

Table 16-28 outlines the calculated geothermal gradient from boreholes drilled between 2020 and 2021. Water samples taken from within the mine shafts during 2024 sinking efforts were consistently near 40°C and in line with geothermal gradient predictions. Groundwater inflow temperature readings taken in 2025 following completion of P/S sinking were recorded at 45°C and also in line with geothermal gradient predictions. The mine is equipped with sinking ventilation fans and a 1.5 MWR chiller to keep temperatures down, with recorded V/S shaft bottom air temperatures entering at 23.5°C wet bulb and exiting at 28.4°C wet bulb.

Table 16-28: 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**

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.

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 3.3 MW. A manual calculation was also carried out based on airflow distribution between leach level which estimated 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.

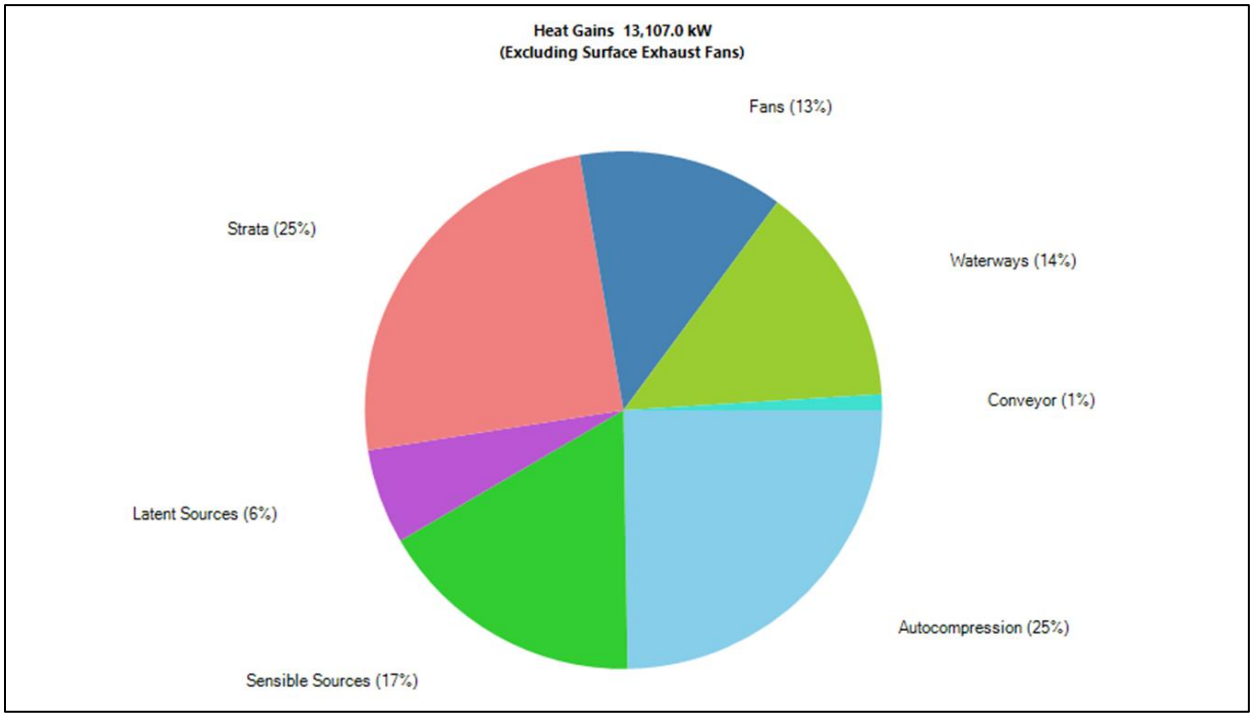
16.8.4.2.6 Ground Water

There are several known water strikes and permeable lithologies from which the KDM UGP will experience high ground water inflows, from initial development through to the 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 the steady state production phase (Year 2029 in VentSim™) indicates the following heat loads (Figure 16-84). The total heat load on the system in steady state production conditions is 13.1 MW, excluding the surface main return fans (RA Fan 1 and 2). The natural cooling capacity of the intake air (5.2 MW) leaves a required mechanical refrigeration of 7.5 MWR.

Figure 16-84: Heat Load Distribution – VentSim™ Model



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 the 470, 310 and 205L shaft stations. The 150 mm NB galvanized carbon steel pipe will be primarily utilised to distribute fire water beyond the shaft stations, however a section of 200 mm NB CS pipe is required to reduce friction losses along the conveyor drive.

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 the P/S connecting the surface infrastructure to ring main units 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 (either 2 megavolt-ampere (MVA) or 1 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 fixed infrastructure and the development jumbos. Key infrastructure such as the crusher, conveyors, and dewatering systems will utilise standardized MPCs to ensure equipment redundancy on site.

Rack based motor control centers (MCCs) or gully boxes will be constructed near the MPCs in substation cut-outs to provide power to the fixed infrastructure and mobile equipment. Bespoke MCCs will be required to support key infrastructure such as the crusher, conveyors and dewatering systems. 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 South African UG mines, as outlined below in Table 16-29.

Table 16-29: 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 (2024)

The electrical reticulation system will support a coincident peak load estimated at approximately 17.8 MVA in Q4 2027. The apparent power has been estimated to be 8.9 MVA during the same period of maximum demand.

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 Battery Electric Vehicle (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.

A portion of the diesel power generation plant (8 MW) used during the initial shaft sinking activities remains on site and provides back-up power generation to site in case of emergency.

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 tele remote equipment UG;
3. A leaky feeder 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 Tele-remote Communication System Requirements

The UGP will potentially utilise a fleet of up to 10 simultaneously tele-remote operated pieces of equipment, including up to 4 LHDs, 5 long-hole drills, and a rock breaking station. 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.

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 considered, a pLTE solution has been proposed for the UGP.

The proposed pLTE solution utilises a leaky feeder cable as an antenna to distribute signal throughout the tele remote area. This provides a better coverage area, as well as more predicable data rates and latency. Equipment data is secured locally and is 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-country and regionally, engaging the Botswana Communications Regulator Authority (BOCRA) for spectrum licences 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, tele-remote operations, and CCTV monitoring.

16.8.8 Compressed Air

The existing compressed air system comprises of 11 x 45.6 m^3/min Ingersoll Rand RM220i-A10 screw compressors providing a total 500 m^3/min at 10 bar psi. Peak compressed air demand occurs during shaft sink when both shafts are blowing over after a mucking cycle and the VSM muckers are utilised to assist. Compressed air requirements will decrease in the subsequent phases of the project, with 5 compressors expected to remain operational in the production phase.

All 11 electric compressors are linked to a common header on surface. The header feeds two 200 NB carbon steel air lines down each shaft, such that each shaft can receive the full system capacity if needed.

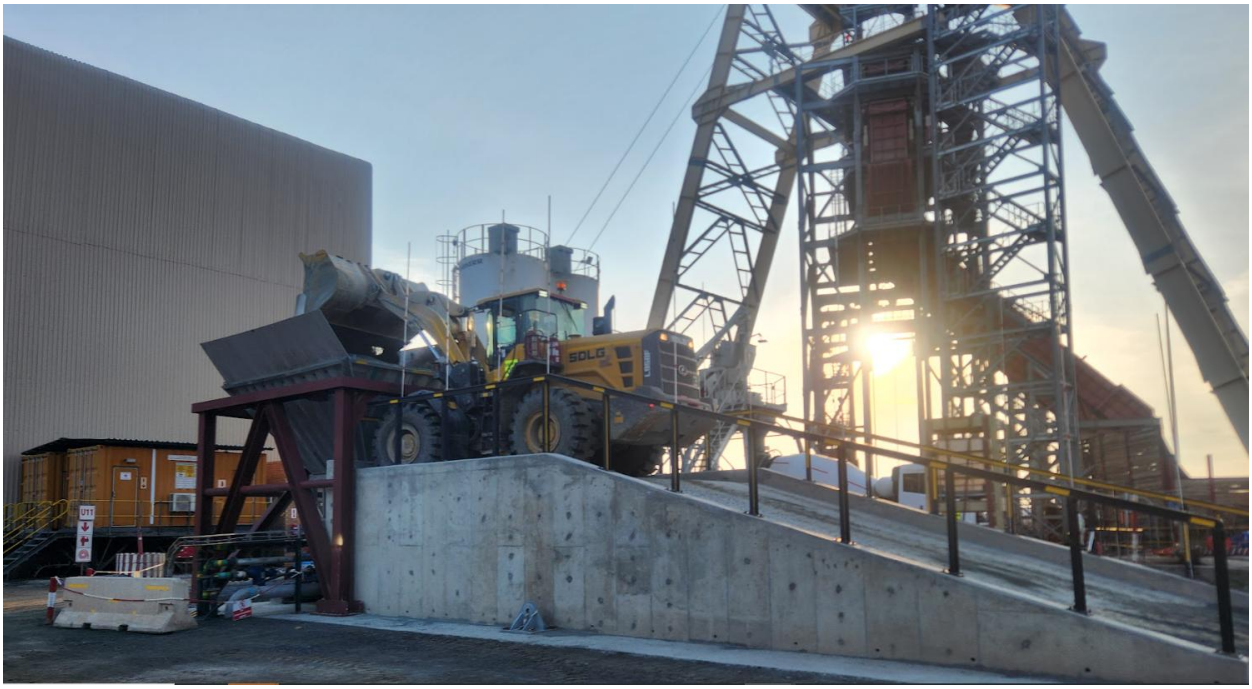
The surface and shaft compressed air reticulation will provide approximately 50 m^3 of air storage. No additional receiver tanks are installed or planned. Only the In-The-Hole drilling equipment is expected to require the mine compressed air system, where additional booster compressors increase pressure from approximately 8 bar to 24 bar. Other mobile mining equipment will be provisioned with onboard booster compressors and/or tanks to provide air pressure or volume as 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 Concrete Batching and Delivery

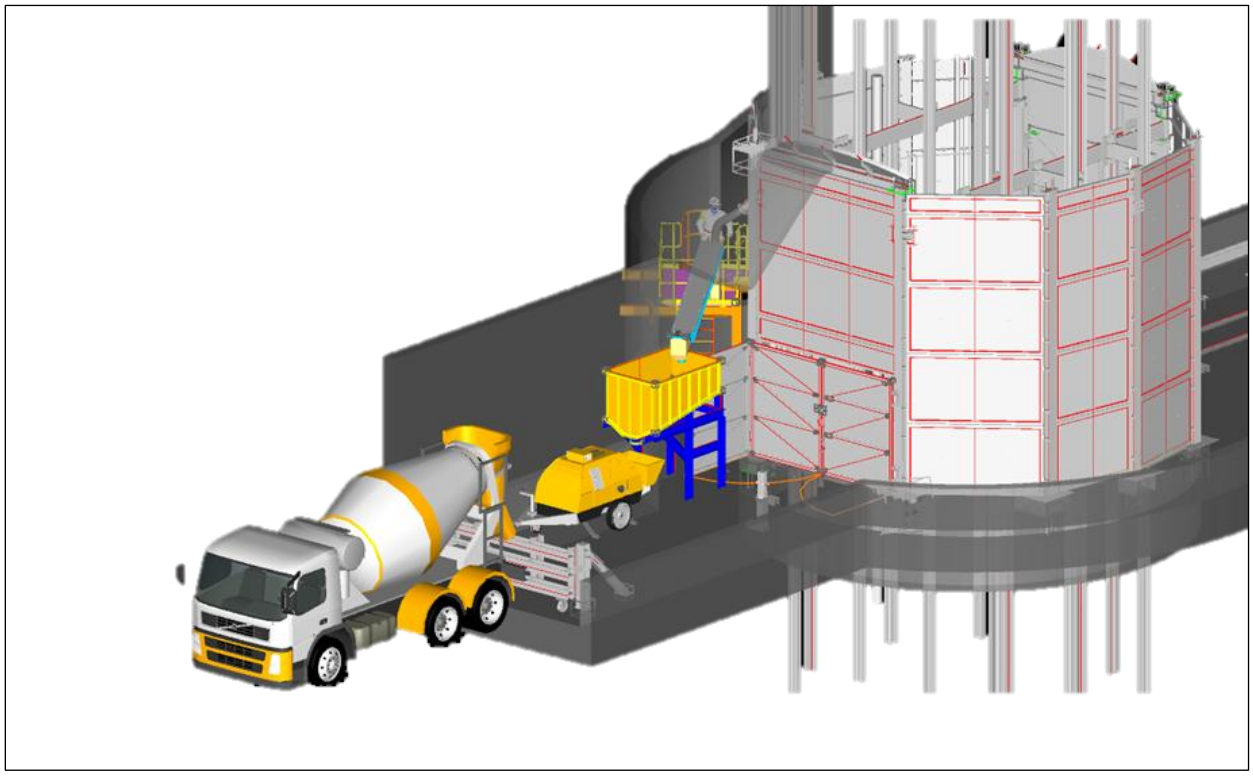
KDM operates two concrete batch plants adjacent to the mine shafts. Batch plants are fed by front end loaders, and mixed product is transported to one of two 150NB shaft slicklines via surface transmixer. Shaft slicklines will deliver wet mix to the 470 lvl and 310 lvl shaft stations, reporting to an UG launder bin with approximately 2.0 m^3 capacity. The bin will gravity-feed a stationary concrete pump which will pump the wet mix to destination or into a 4.0 m^3 capacity UG transmixer. Dual slicklines will allow simultaneous delivery to two different shaft stations if needed and can be interchanged during periods of maintenance. The station bins allow for quick discharge of concrete from the slicklines, reducing risk of concrete setting within the shaft barrel. Station development will be graded to allow slickline clean-out to a sump located adjacent to the transmixer loading area, where transmixers will also be washed. Shotcrete will be batched and delivered underground using the same systems. Figure 16-85 shows the batch plant feed hopper and Figure 16-86 shows the UG launder bin and concrete pump arrangement.

Figure 16-85: Batch Plant Loading



Source: Lucara (2024)

Figure 16-86: Concrete Delivery System Underground



16.8.10 Explosives and Detonator Storage

The KDM UGP mine plan will the following explosive products:

- Bulk Emulsion;
- Packaged Emulsion;
- Detonators (including wireless);
- 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 personnel and material cage. UG, the totes will be collected and transported to one of two (2) 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 utilised 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 considered peak requirements for both development and production and have been sized to accommodate one week's worth of consumption. Capacities are identified in Table 16-30 and Table 16-31 below. Note that not all magazines will store these volumes at once and wireless detonators will generally not be stored underground as they are programmed on surface for each blast before reporting underground. 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. UG magazines maintain 100 m offsets from shafts and stopes, and at least 10 m between Type A and Type B explosive magazines.

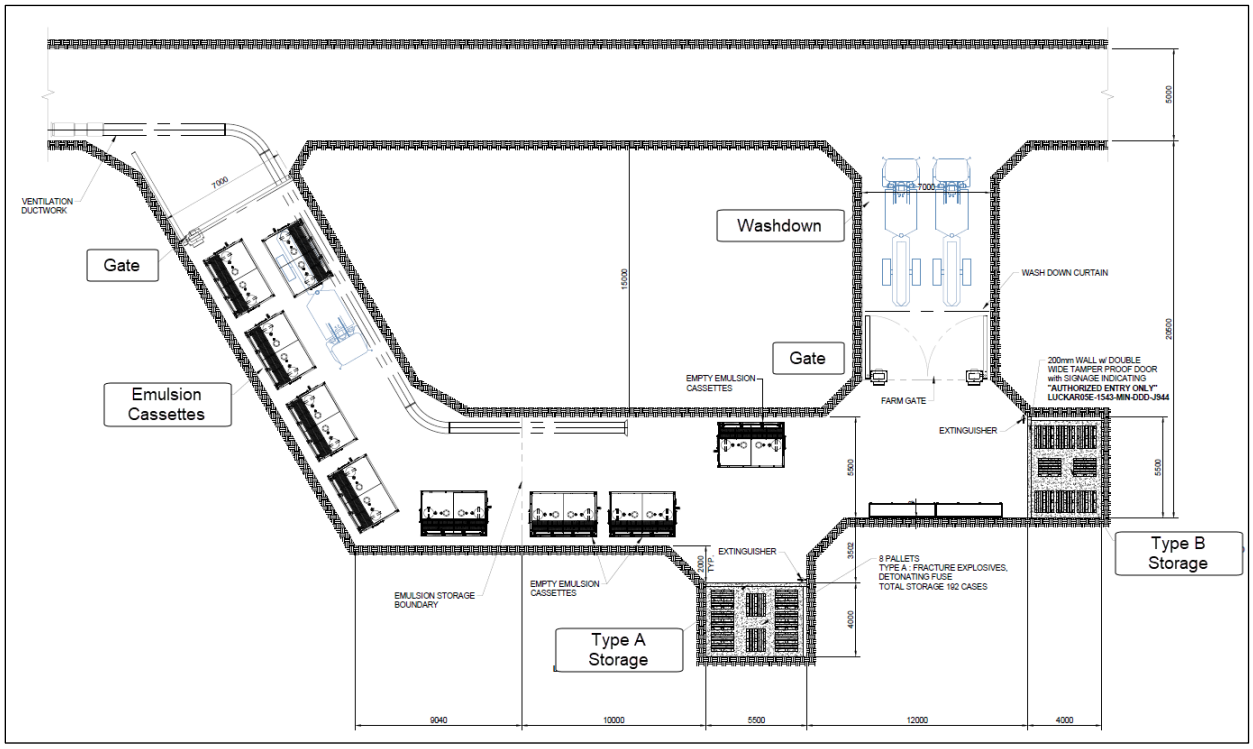
Table 16-30: Magazine Capacity 470 Station

Item	Type	Quantity (kg or Units)	Cases/Bags
Emulsion	Oxidizer	72,000	24 Totes
ANFO	Explosive	8,000	350 Bags
Packaged Emulsion	Explosive	2,000	100 Cases
Detonator	Explosive	2,000	100 Cases

Table 16-31: Magazine Capacity 310 Station

Item	Type	Quantity (kg or Units)	Cases/Bags
Emulsion	Oxidizer	38,000	24 Totes
ANFO	Explosive	4,000	350 Bags
Packaged Emulsion	Explosive	4,000	100 Cases
Detonator	Explosive	4,000	100 Cases

Figure 16-87: Magazine General Arrangement



16.8.11 Fuel Storage and Distribution

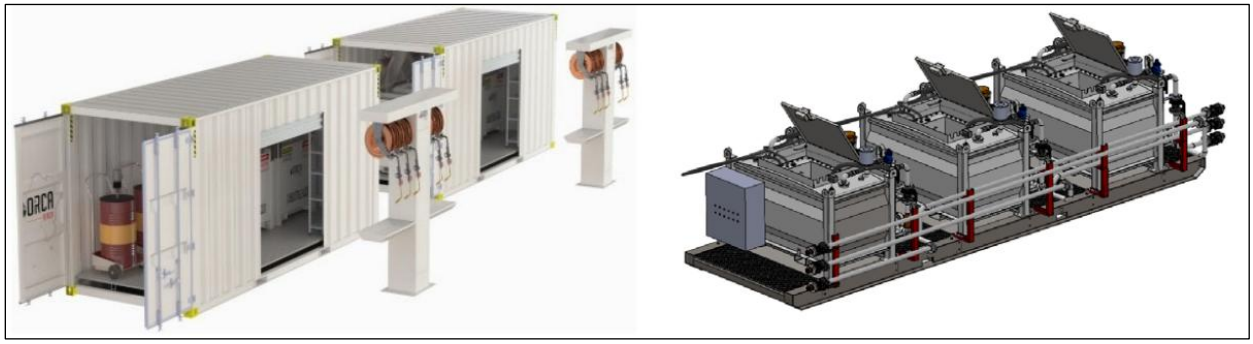
An equipment fueling and lube station will be located near the shafts on 470 lvl and will be able to provide fuel for the mobile UG equipment fleet. On the 310 lvl, a larger fueling and lube station will be located near the drawpoints to provide quick access for the production LHDs.

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 utilised to minimise construction efforts and allow portability of equipment, as illustrated on the left of Figure 16-88. Pumps will be pneumatically operated and reversible to allow simple and efficient transfer of fuel between transport totes and the dispensing tanks.

Lubrication equipment, discussed in Section 16.8.12 will be skid mounted as illustrated on the right of Figure 16-88 and require external bunding and fire protection.

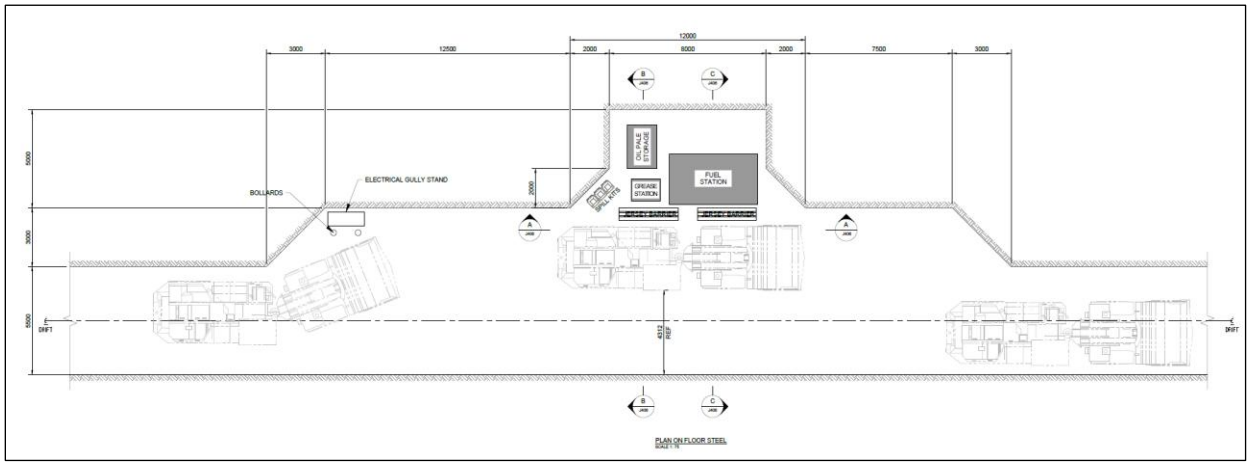
Fuel bays will be developed as drift slashes, minimizing the need for dedicated ventilation and allowing for equipment to queue without obstructing traffic, as illustrated in Figure 16-89.

Figure 16-88: Fuel and Lubrication Equipment



Source: Orca (2025)

Figure 16-89: Fuel Bay General Arrangement



16.8.12 Mobile Equipment Maintenance

A main underground maintenance facility will be built at 310 lvl for equipment servicing and repairs, with smaller facilities at 470 lvl for minor work. The main shop will be located near the extraction area. Ventilation is supplied by the 310 lvl P/S, with a dedicated exhaust raise connected to the 285 lvl return air circuit for flow-through ventilation. Fire doors will regulate airflow under emergency conditions.

The workshop will accommodate up to six (6) machines at once during peak commercial production. This is calculated based on a fleet of 22 equipment units and 26 light vehicles, as well as operating forecasts and service intervals detailed in Table 16-32. Figure 16-90 illustrates the general arrangement of the proposed workshop, which can be seen on the level plan in Figure 16-57. 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, as is current practice during shaft station development.

Figure 16-90: Workshop General Arrangement

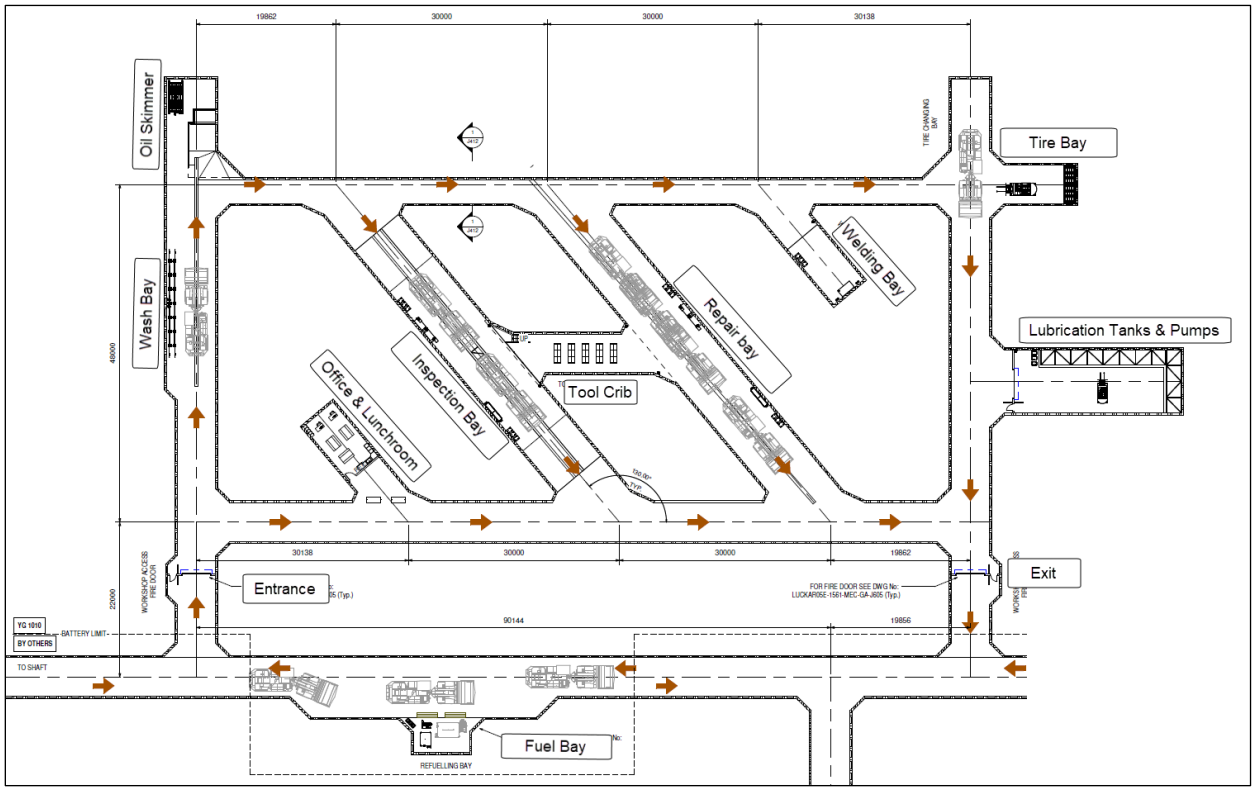


Table 16-32: Workshop Sizing Assumptions

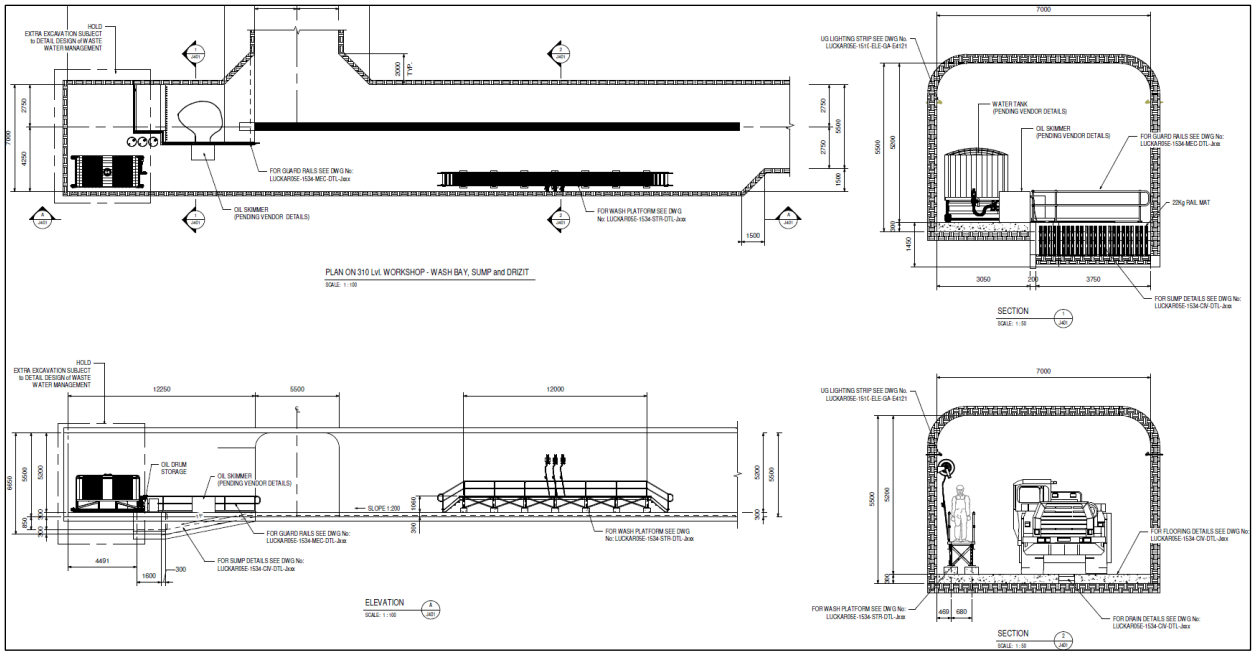
	Assumption	Quantity	Comment
Available Work Time	Work days per week	7	Monday - Sunday
	Maintenance shifts per day	1	Night Shift on call-out
	Shift Efficiency	75%	9/12 hr shifts
Major Services	Services per 2 weeks (minimum)	1	
	Equipment hours between services (maximum)	125	
	Major service duration (shifts)	1	
Minor Services	Services per 2 weeks (minimum)	2	
	Equipment Hours Between Service (maximum)	24	

	Assumption	Quantity	Comment
	Service Duration (Shifts)	0.5	
Unplanned Maintenance	Unplanned repairs (ratio)	40%	
	Unplanned service duration (shifts)	5	
Mechanical Availability	Loaders, Drills, and Aux Fleet Target M.A.	80%	

The workshop design considers continuous, safe and process-oriented flow. The Maintenance Area will be one-way traffic and will have a single entrance and exit. Equipment will come from the extraction zone and 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 return to the working face.

The wash bay, illustrated in Figure 16-91, will be equipped with sump and wastewater treatment system. This sump will be the primary collection point for all wastewater generated in the workshop. Oily water will be filtered with oil reporting to drums for safe handling, and clean water reporting to main sumps for discharge to surface treatment ponds.

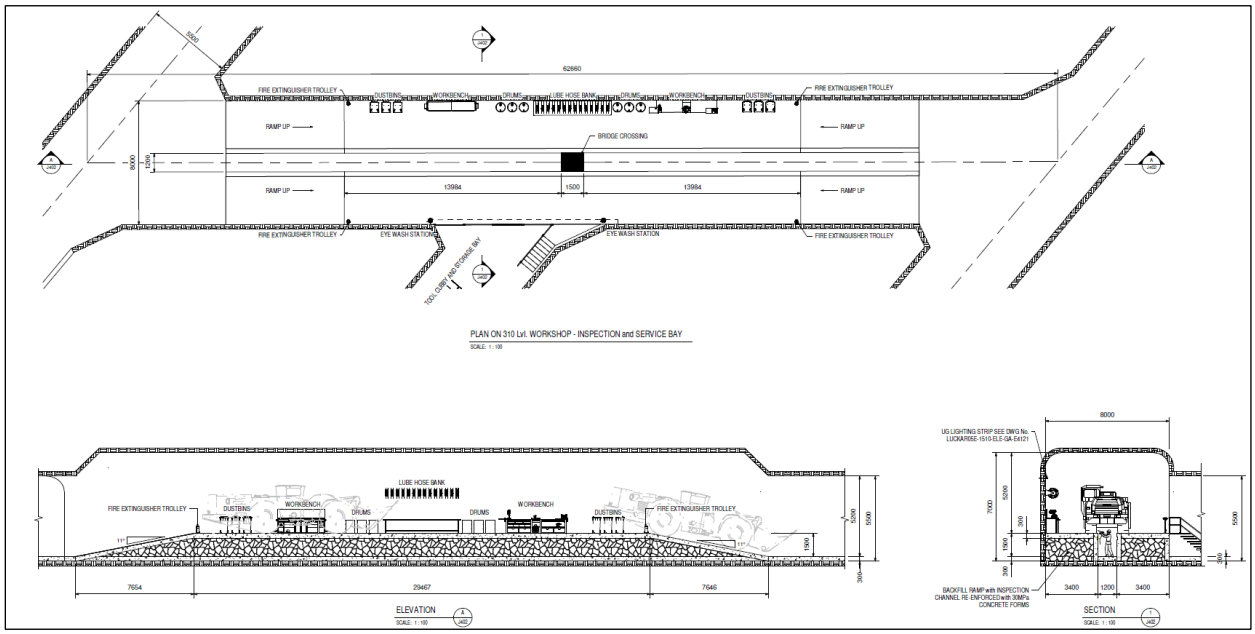
Figure 16-91: Wash Bay General Arrangement



The Repair Bay is designed to accommodate 2-3 machines at a time and supported with a 15 t overhead crane. The floor will be sloped to a central drain equipped with sump and pump to manage spillage. Standard safety equipment including fire suppression, first aid, eyewash, and spill kits will be provisioned.

The Inspection and Service Bay will be equipped similarly to the Repair Bay, but without an overhead crane and instead a drive-through inspection ramp suitable for servicing two machines. Between the two bays is a central tool cubby.

Figure 16-92: Inspection and Service Bay General Arrangement



Hose and cable reels will be mounted in both bays 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 will 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. Fire doors with side-entry personnel doors will be equipped at each entrance and exit.

An Environmental Station will 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-organised environmental station improves handling and reduces risks by centralizing the waste storage for the facility.

The workshop will house, in addition to the above, a welding bay, tire bay, office, and warehouse.

16.8.13 Mine Safety

The P/S station, being in fresh air, will serve as the primary muster point and point of refuge underground during a mine emergency. Beyond 300 m from the shaft, underground workers will report to refuge chambers established throughout the mine.

Two permanent refuge stations will be located on the 310 lvl, with an eating area, ablutions facility, storage and parking located adjacent to the facility. Additional self-contained portable refuge stations will be located on the 285 lvl, 310 lvl, 340 lvl, 405 lvl, 470 lvl, and 580 lvl. 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.

Automated fire suppression 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. All UG heavy equipment will be equipped with automatic fire suppression systems. Every vehicle will carry at least one fire extinguisher of adequate size.

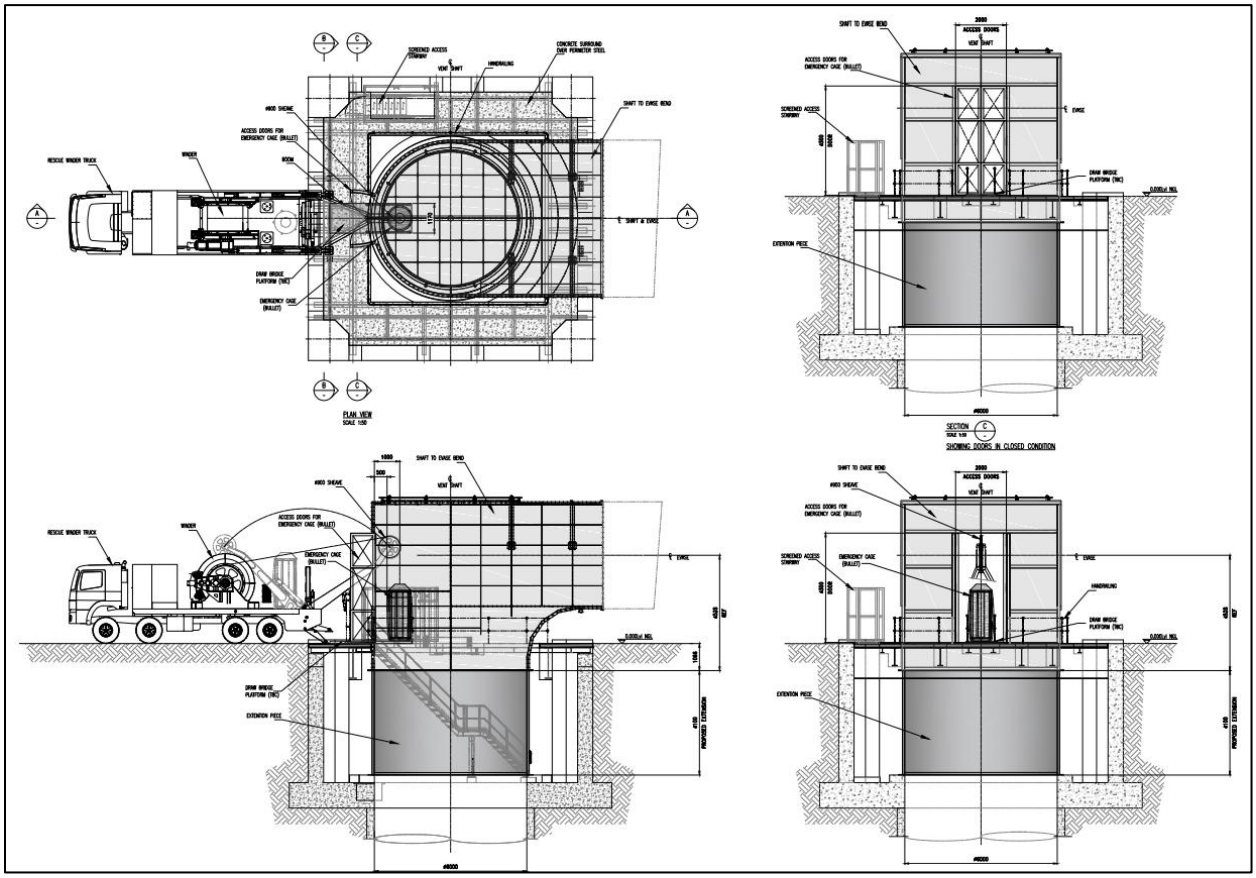
A fully equipped mine rescue team will be available every shift to respond to emergencies.

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.

16.8.14 Mine Egress

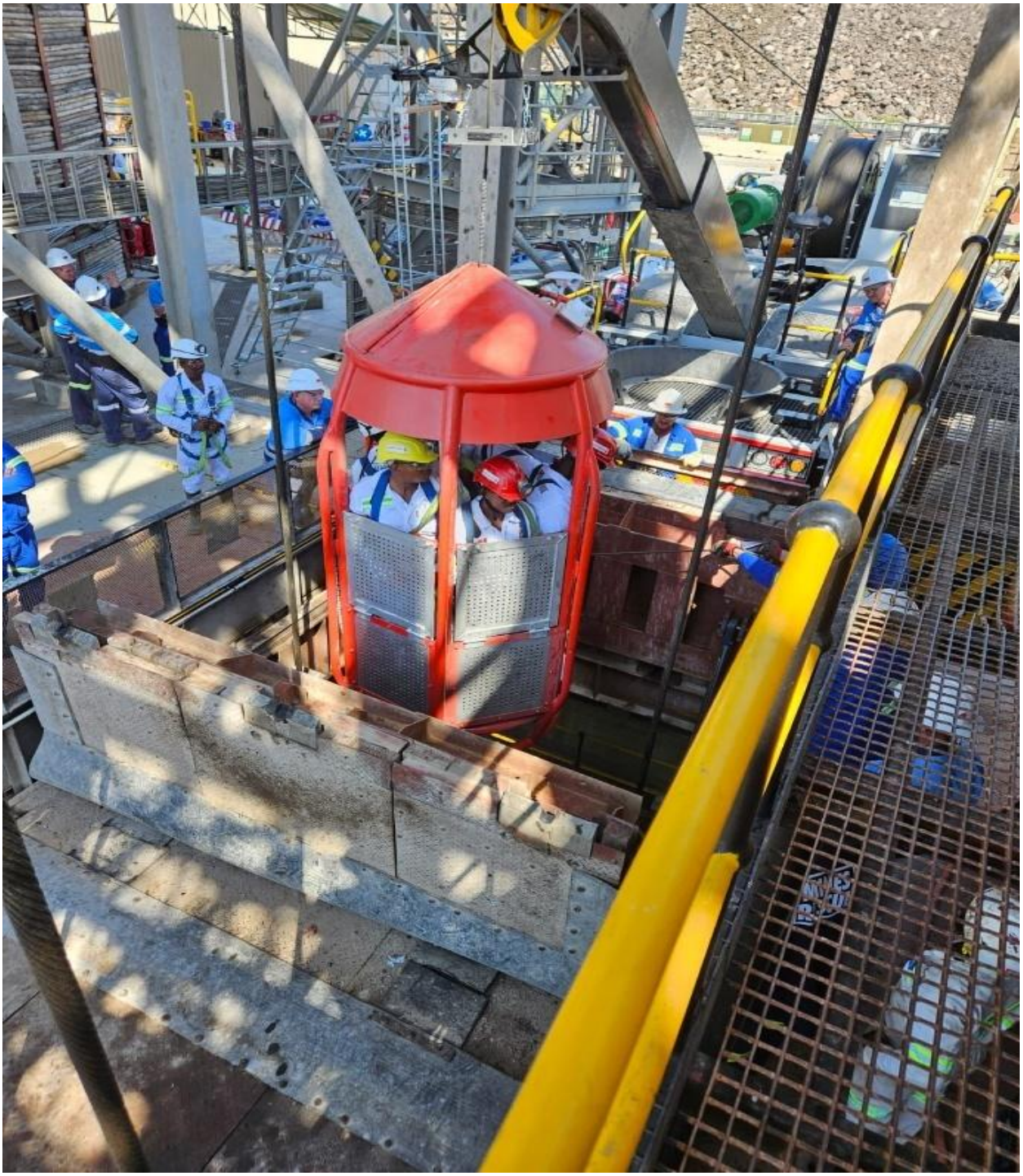
No ramp or ladderway is planned to connect the UG mine to surface. Primary mine access will be through the P/S via either the Man and Material Cage or auxiliary cage. Secondary emergency egress will be via a mobile emergency mobile bullet winder procured and commissioned on site in 2023, seen in Figure 16-94. In the event of an emergency in which the P/S is unable to hoist people to safety, the mobile winder will back up to the shaft and deploy a bullet with 6 person capacity through the VS as illustrated in Figure 16-93. As part of the emergency evacuation procedure, the main surface fans will be shut down (if running) and emergency bulkhead fans located at each shaft station will be turned on to ensure fresh air flow between Shafts.

Figure 16-93: Mobile Emergency Winder Deployment



Source: UMS (2022)

Figure 16-94: Mobile Emergency Winder Commissioning 2023



Emergency scenarios were simulated in 2022 to inform mobile winder design criteria and general egress mine designs. Absolute worst-case events that have the potential to render the P/S cages unavailable and call for the use of the mobile winder were evaluated to determine the winder size class required.

A list of possible emergencies was created, ranging from equipment fires to medical issues. Only scenarios that the primary egress system could not handle were selected for further review. These include:

1. Mechanical failure: Severe shaft incidents can damage critical components and electrical systems, rendering cages, skips, and underground mine power inoperable;

2. Power failure: Extended national grid outages and diesel backup failures can stop power supply to the underground mine; and

3. Fall of ground: Major collapses in the shaft may destroy essential infrastructure and cut off power, leaving cages and skips unusable.

In all scenarios the bottleneck to evacuation time was the battery life on portable refuge chambers, which typically does not exceed 48 hours. It was evaluated that for a workforce of 156, a 6-person bullet moving at 1 m/s can evacuate everyone in 32 hours, which is well below the 48 hour battery provided in most portable refuge chambers. As contractors, workplans, and staff loading is updated over time this evacuation strategy and emergency evacuation management plans will be updated.

16.9 Unit Operations

16.9.1 Drilling

Development headings will be developed by a combination of split-feed and solid-feed, two-boom jumbo drills. Jumbos will be equipped with either 5.5 m (18') solid-feed or 4.3 m (14') split feed configurations varying by working area. Jumbo advance per heading is budgeted between 1.3 – 4.5 m/d per machine depending on profile size and ground support requirements. After the P/S is equipped, development metres ramp up from 250 m/month to a maximum of 800 m/month.

Production drilling will be performed by six (6) ITH drills capable of drilling >100 m long 165 mm (6.5") production holes. Smaller diameter holes will be used for drawbells, undercut, and other uphole blasthole drilling. Since drills operate on multiple levels and need to be moved between horizons, most of the ITH drills will be track mounted instead of carrier mounted. This keeps them compact and light enough for cage transport with only the boom detached. The Man and Material Cage has, in turn, been designed to accommodate a DU311TK drill in this configuration. The ITH drills will operate with skid-mounted rod handlers to allow for long hole drilling, where required, in excess of the standard carousel capacity (typically 30-40 m). Tow-behind compressors will boost mine air to that required for large production holes and slot reamed holes. A V30 reamer head will allow for slot holes of 760 mm to be drilled. A carrier mounted ITH drill with similar capacity will also be used to drill off drop raises, drawbells, and undercuts on the lower mine levels.

16.9.2 Blasting

Bulk emulsion will be used for production blasting and development rounds. Bulk emulsion will be non-sensitized until gassed at the face. Boosters, primers, detonators, detonation cord and other ancillary blasting supplies will also be utilised. Smooth blasting techniques may be used as required in headings, with the use of trim powder for loading the perimeter holes. Wireless detonators will be employed in production rings and will play a critical role in the mine's ability to tight fill the cave for air blast prevention, which requires charged blast rings be buried under run-of-mine material. Project budgets allow for on-site support by the explosives manufacturer for blast planning, calibration, reconciliation, and management of wireless detonator hardware and software.

Blasting shall be conducted at end of shift and has been factored into cycle times. Production blasts shall be grouped together where possible to minimise extended mine clearance. Ventilation system design considers opportunities to keep certain mine infrastructure staffed and operational (i.e., crusher) during blast clearance.

16.9.3 Ground Support

Primary ground support will be installed post-mucking of the blasted drift. Secondary and tertiary support may be installed out of the development cycle by the service crew in accordance with the Ground Control Management Plan.

Electric-hydraulic split-feed jumbos will be used primarily for support installation, with cable bolters, jacklegs, and other tooling used where required for secondary support.

A mobile shotcrete sprayer will apply wet mix product, which will be batched on surface and transferred underground via slicklines before being transported to the work face via transmixer.

Regular pull tests will be conducted on-site to ensure adequate installation of resin rebar, split set, and cables bolts. Shotcrete, when required, will also be sampled by use of splatter boards and in situ coring, and sprayed panels, to be tested for strength and adequacy in accordance with the GCMP and QA/QC requirements.

16.9.4 Mucking

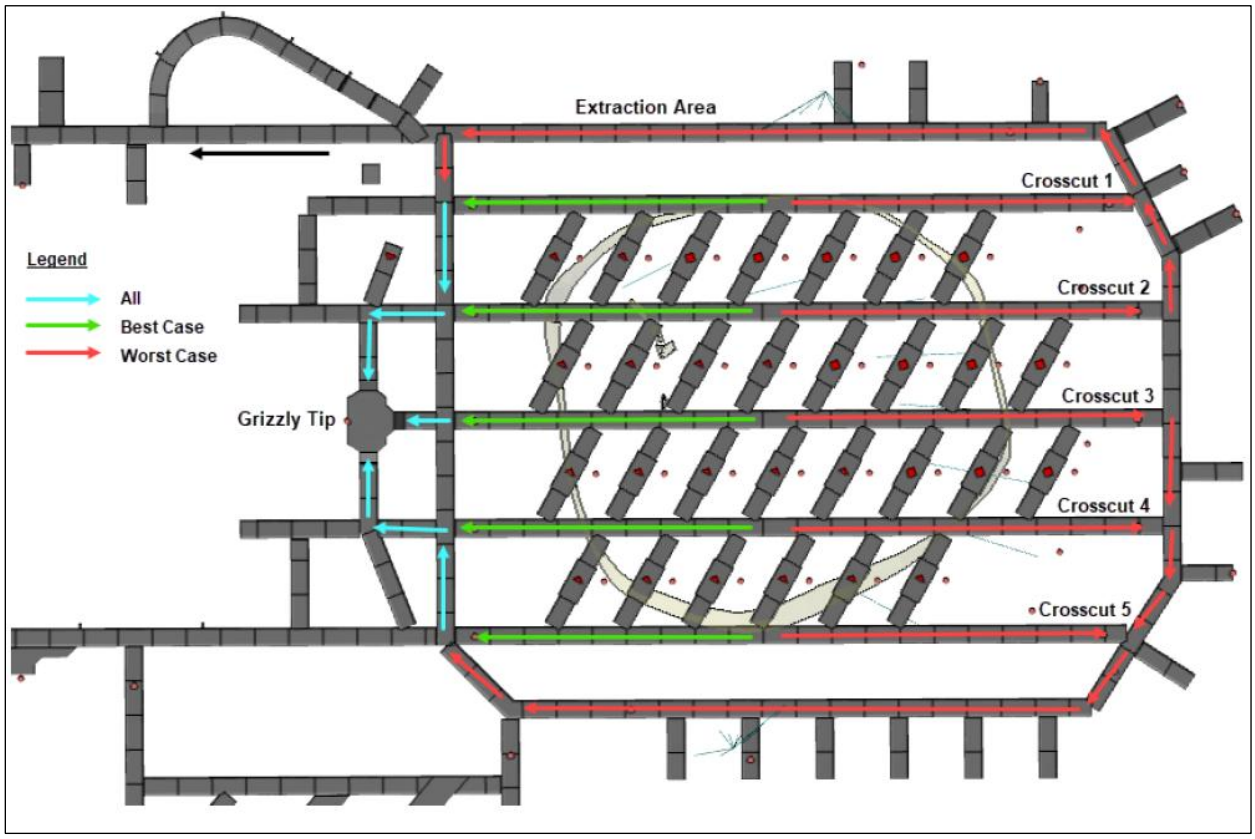
Both 17 t (8.4 m^3) and 21 t (9.5 m^3) nominal capacity LHDs will be used for lateral development. Rock will be trammed from the face and either temporarily stored in a remuck or directly trammed to a rock pass.

Production ore will be mucked from drawpoints 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 will later rehandle this material when space becomes available.

LHD cycle times and quantity requirements were calculated from first principals. Several Arena simulations were performed between 2019 and 2021 (refer to Section 16.7.3.1) to affirm the

extraction level selection and LHD fleet size requirements, which concluded the need for 3-4 machines to sustain production levels. In 2025 additional cycle analysis was performed to repeat this exercise on updated mine designs and to incorporate worst and best case haul cycles during periods of drawpoint maintenance. A best case haul cycle is one where the LHD trams directly to the tip or drawpoint, whereas a worst case haul cycle is one where there is an obstruction in the extraction drive which requires the LHD to tram around the perimeter drives, as illustrated in Figure 16-95.

Figure 16-95: Best and Worst Case Haul Cycle



Key operational assumptions used in the analysis included 59 drawpoints with an overall availability of 73%, 15 hours of effective work per day per LHD, 5 non-operational days per year, and a maximum of one LHD operating in a single panel at a given time. Drawpoint availability takes into consideration hangup frequency once every 1,000 tonnes drawn, oversize once every 500 tonnes drawn, and a roaming extraction crosscut rehabilitation crew which has half of an extraction panel down for repairs at all times.

Two Sandvik Load-Haul-Dump (LHD) models were evaluated: the LH621 (21 t / 9.5 m³) and the LH517 (17 t / 8.4 m³). It was found that both models can meet production requirements, but the

LH621 offers a 14% higher average mucking rate, which is advantageous during periods of intensive rehandle.

Cycle time analysis details are highlighted in Table 16-33 for an LH517 only scenario, which indicates an average 6.2 minute cycle and the need for 3.6 LHDs to meet production rate. For a LH621 scenario, 3.1 machines are required.

Table 16-33: Haul Cycle Analysis

Steady State Extraction Haulage	Units	Extraction Panel					Total
		1	2	3	4	5	
Drawpoints Constructed	#	6	14	16	15	7	**58**
Panel Availability (Scheduled Maintenance)	%	90	90	90	90	90	
Drawpoints Available After Panel Maintenance	#	5.4	12.6	14.4	13.5	6.3	**52.2**
Chance of Hang Ups or Oversize	%	20	20	20	20	20	
Drawpoints Down for HU or OS	#	1.09	2.54	2.90	2.72	1.27	**10.51**
Remaining Drawpoints Available	#	4	10	12	11	5	**42**
Drawpoint Availability	%	72	72	72	72	72	**72**
Target draw per day per Drawpoint	t/d	190	190	190	190	190	**190**
Target draw per day per Panel	t/d	819	1,911	2,184	2,047	955	**7,917**
Best Case Cycle Time (Avg per panel)	seconds	344	329	300	329	348	**348**
Worst Case Cycle Time (Avg per panel)	seconds	508	529	534	547	532	**532**
Applied Cycle Time	seconds	363	355	331	357	370	**370**
	minutes	6.1	5.9	5.5	6.0	6.2	**6.2**
Available working hours per day per LHD	hrs	15	15	15	15	15	**15**
Selected Bucket Size	t	17	17	17	17	17	
Maximum Potential Mucking Rate	t/d	2,528	2,590	2,777	2,572	2,483	**12,950**
Actual Mucking Rate (caps at Target if met)	t/d	819	1,911	2,184	2,047	955	**7,917**
Actual Mucking Hours	hrs/day	4.9	11.1	11.8	11.9	5.8	**45**
Equipment Availability	85%						
LHDs Required	**3.6**						

16.10 Mine Personnel

Mine development contractors will be utilised for mine construction and pre-production operations. The mine plan envisions, for budgetary purposes, five separate mine development contractors; one each for the following:

- Shaft sinkers;
- Development Crews;
- Construction Crews;
- Electrical Crews; and
- Production drillers.

Several existing OP contract services will continue to support UG operations, including surface load and haul, and site road maintenance. Other contractors or OEM vendor support will be engaged as necessary to oversee installation or commissioning of infrastructure or services.

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 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 care and maintenance, including rehabilitation, of UG workings. The Owner's Team is organized into the following departments:

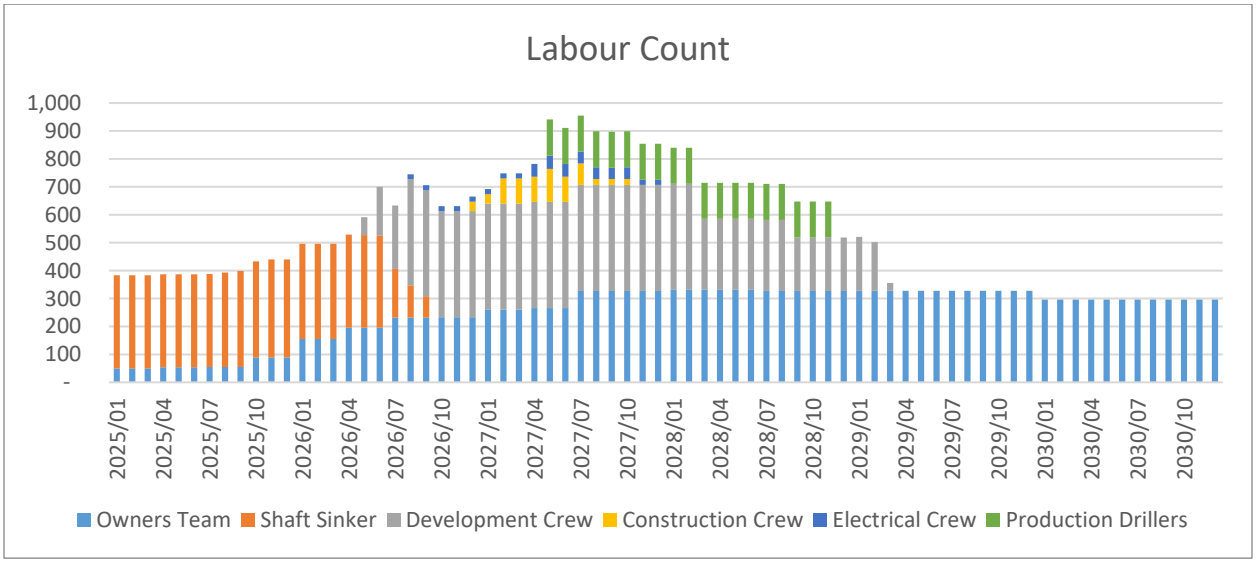
- Projects / EPCM;
- Technical Services (Engineering, Planning, Geology, Survey);
- Mining (Shaft and Drawpoint Operations);
- Mine Maintenance (Shaft, Surface, UG Fixed, Mobile Equipment); and
- Mine General (Finance, IT, Human Resources, Procurement, Safety, Risk).

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 6x3 schedule or equivalent. During mine operations UG labour will work a 4x4 schedule. Management, technical services, and contractor supervisory roles will work a 5x2 schedule where appropriate.

Total required mining labour up to steady state operations in 2030 is summarised in Figure 16-96. Peak project labour is estimated at 955, with a peak on-site dayshift of 553 in 2027.

Figure 16-96: UG Labour Force



16.11 Mine Equipment

The mobile equipment fleet for the UGP is diesel-powered, trackless, and rubber tired. The contractor performing lateral development and production drilling at the UGP will have their mobile equipment supplied by the Owner. This approach reduces costs and ensures equipment is owned by the Owner at the conclusion of the lateral development contractors scope to perform drawpoint operations and mine maintenance activities. The contractors performing underground construction will be responsible for supplying all mobile equipment required for their scope.

UG equipment requirements are built up from contractual requirements for lateral development equipment or productivities for unawarded contracts and production activities. As such, equipment requirements fluctuate throughout the mine life. Major equipment productivities used to estimate equipment requirements are as follows:

- Production drilling: 11 m/hr or 390 t/hr; and
- Production mucking: 175 t/hr.

Owner supplied equipment requirements for both the mine development contractor, production drilling contractor and drawpoint operations activities are summarised in Table 16-34. Contractor supplied underground construction equipment utilised primarily in 2027 has been excluded.

Table 16-34: Mobile Equipment Requirements

Equip Type	Max	2025	2026	2027	2028	2029	2030
Drill & Blast							
Jumbo - 2 Boom - Small	5	2	5	5	5		
Jumbo - 2 Boom	2		2	2	2	1	1
Cable Bolter	1		1	1	1		
Long-hole Drill - ITH (Dev)	2		1	2	2		
Long-hole Drill - ITH (Prod)	4			4	4		
Small Explosives Truck	2		2	2	2		
Large Explosives Truck	2			2	2		
Load and Haul							
LHD (7 t / 2.8 m^3)	1	1	1	1	1	1	1
LHD (12.5 t / 5.5 m^3)	1	1	1	1	1	1	1
LHD (17 t / 8.4 m^3)	4		4	4	4	4	4
LHD (21 t / 9.5 m^3)	2		2	2	2	2	2
Ancillary Equipment							
Telehandler UG	4	2	2	4	4	2	2
Integrated Tool Carrier	3		2	3	3	2	2
Boom Truck	3		1	3	3	1	1
Transmixer	2		2	2	2	1	1
Shotcrete Sprayer	1		1	1	1	1	1
Shotcrete Cassettes	1		1	1	1		
Secondary Breakage Drill	1		0	1	1	1	1
Rock Breaker Attachment	2		0	2	2	2	2
Mini Excavator	4	4	4	2	2	2	2
Fuel/Lube Truck	1		1	1	1		
Service Truck	2		2	2	2	1	1
Utility Vehicle	17		2	17	17	13	13
Surface Equipment							
Telehandler 24T	2	2	2	2	2	2	2
Telehandler 10T	1	1	1	1	1	1	1
Warehouse Forklift	1	1	1	1	1	1	1
Concrete Mixer Truck	3	3	3	1	1	1	1
Front end loader	1	1	1	1	1	1	1
Tractor	1	1	1	1	1	1	1
Combi - 16s	2	2	2	2	2	2	2
Utility Vehicle - Surface	7	7	7	7	7	7	7
Hiab Loader Crane	1	1	1	1	1	1	1

16.11.1 Electric Mine Fleet

A diesel LHD fleet should be used to initiate production activities at the UGP 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 the UGP 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 and equipping schedule was provided by the shaft sinking contractor. The QP completed the remainder of the development and production schedule.

The project schedule looks forward from Q4 2025 and consists of a three-year pre-production period and a 10-year operating period.

Shaft sinking commenced in Q3 2021 and will be completed in Q4 2025. P/S equipping is scheduled to complete by Q3 2026. Lateral development begins once the shaft sinking is complete. Production ramp up begins Q4 2027 with production commencing in Q1 2028.

16.12.1 Scheduling Philosophy

Both shafts have been sunk in parallel and are entering equipping phases with lateral connections established between shafts on most stations. During P/S equipping the V/S will provide personnel and material access to the skip loading station for construction purposes, as well as supply ventilation underground via shaft ducting while the P/S is stripped of its sinking duct.

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 following the commissioning of the P/S on the 310 lvl and 285 lvl stations where the lateral development equipment used by the shaft sinking contractor 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 lvl flood drift and the 310 lvl Northern Drive towards the ore body, which will be connected by vertical raises to establish a ventilation circuit;
- Ramp to P/S bottom, as to establish access for mucking out of shaft spillage; and
- Ramp to 335 lvl from 310 lvl to enable excavation and construction of the fine ore bins.

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 lvl, 405 lvl, and 470 lvl sub levels; 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 lvl) and end at the top of the mine (670 lvl). As development fleets become available, they will be relocated to upper mine levels to develop future production horizons.

16.12.1.1 Pre-Production Sequence

Section 16.7 discusses the design and arrangement of the extraction level, undercut, and stoping horizons, which will be developed in the following sequence:

1. 340 lvl Undercut drill drives are developed across the ore body from west to east;
2. A 2.4 m x 2.4 m square slot raise is drilled at the end of each drill drive using an ITH drill and V30 reamer;
3. Uphole fan-shaped rings (25 m high x 31.5 m wide) are drilled in each drive retreating from east to west;
4. Starting in the northeast corner of the Undercut level, the slot raise and rings are blasted in retreat from east to west;
5. Swell tonnes (36%) are mucked from the Undercut drill drive;
6. A 2.4 m x 2.4 m square slot raise and additional blast holes are drilled from the Extraction Level and blasted to create each drawbell. Drawbells can only be fired once the Undercut above has advanced at least 20 m past the drawbell, creating a stress shadow that minimises damage to the draw points below;
7. Swell tonnes (36%) are mucked from the bottom of the drawbell;

8. Long-hole shrinkage (LHS) stopes are sequentially drilled and blasted above the broken undercut and drawbells and again with swell of 36% drawn via the drawbells; and

9. Upon cave initiation draw progressively ramps up in the respective drawpoints, while managing airgap and encouraging cave initiation across the whole footprint.

16.12.1.2 Drawbell Construction

Drawbell construction rates are a key parameter for determining the extraction level and undercut establishment and ramp up. In an advanced undercut pre-production sequence, drawbell construction rate often dictates the speed at which the undercut can be advanced and therefore is an area of critical importance in the schedule.

Drawbell construction rates at Karowe have been developed using first principles buildup as well as industry benchmarks from other caving projects. On average, drawbells in the Karowe project schedule take 43 days to be completed. This includes drawpoint development, ground support, drawbell drilling, charging, blasting, and mucking. A maximum of three drawbells are constructed in any one-month period.

16.12.2 Mine Development Schedule

Deswik software was used to prepare the mine development schedule.

16.12.2.1 Shaft Sinking Schedule

The shaft sinking schedule was developed in Microsoft Project by the sinking contractor and then transferred into Deswik. Task rates were reverse-calculated from start and end dates provided within the sinking contractor's Microsoft Project Schedule for visual aid purposes.

16.12.2.2 Lateral and Vertical Development Rates

The following scheduling constraints were used in Deswik for all lateral and vertical development:

Distinct development resources are assigned to each captive level:

- 1.3 – 4.5 m/d advance rate depending on heading dimensions and ground support requirements.

Two raise boring crews:

- 2.5 m/day blended piloting and reaming rate.

Lateral development capacity:

- 800 m/month across all levels, ramping up from 250 m/month over a period of 7 months.

Development cycle times and productivities used for mine development were provided as a competitive bid from reputable Southern African mining contractors.

16.12.2.3 UG Infrastructure Installations

UG infrastructure installations have been accounted for within the mine schedule. Table 16-35 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 utilised for infrastructure installations depending on the task and time period.

Table 16-35: Major Infrastructure Installation Durations

Infrastructure	Units	Duration (days)
1510 – Sub Station	Days	21
1510 – Rock Breaker	Days	110
1510 – Crusher Chamber	Days	210
1522 – Conveyor System	Days	210
1540 – Refuge (Portable)	Days	10
1550 – Typical Sump	Days	14
1558 – Pump Room	Days	120
1560 – Ventilation Bulkhead	Days	10
1560 – Ventilation Door	Days	10

16.12.2.4 Mine Development Summary

At KDM, most development work must be finished before production starts. Production stoping requires fully sunk and equipped shafts, along with completed underground dewatering, ventilation, power distribution, and material handling systems. Sustaining mine development will consist of the 580 Instrumentation Level developed from the 670 Shaft Station and the 245 Flood Drives. However, this development will be completed as soon as possible to capitalize on the momentum the development contractor will have developed on site.

16.12.3 UG Production Schedule

The annual mine production schedule provided in Figure 16-97 and Table 16-36 combines development and drill & blast schedules prepared in Deswik with cave draw schedules produced through numerical modelling. These schedules are merged to provide a cohesive annual summary of development, drill & blast, cave propagation, draw, and final hoisted material to surface. Ore and waste tonnages have been rounded to the nearest million.

The schedule includes actual development achieved in 2025 up to end of September, with forecasted mine schedules October onwards.

Figure 16-97: Hoisted Tonnes and Grade by Domain

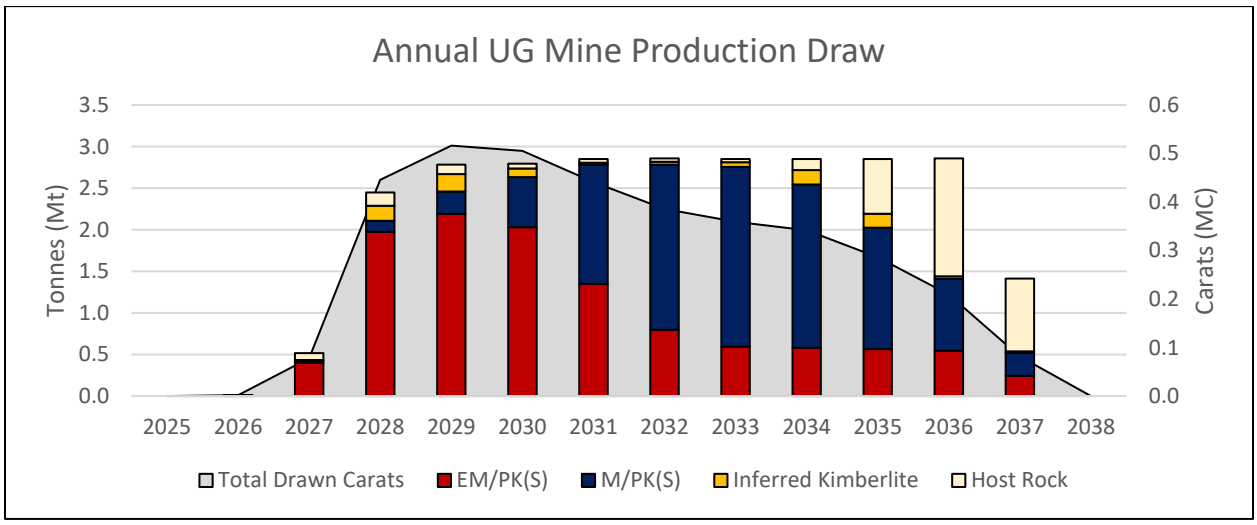


Table 16-36: Summary of UG Mining

Parameter	Unit	Total	2025[1]	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Development																
Shaft Development	km	**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
Lateral Development	km	**16.4**	0.2	2.9	9.4	3.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Internal Raises	km	**0.9**	0.0	0.4	0.5	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Drill & Blast																
Development Ore	Mt	**0.5**	0.0	0.0	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Drawbells	Mt	**0.2**	0.0	0.0	0.0	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Undercut	Mt	**1.0**	0.0	0.0	0.3	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Stoping	Mt	**7.7**	0.0	0.0	0.2	6.4	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Blasted	Mt	**9.4**	0.0	0.0	0.8	7.5	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Production Draw																
Development and Undercut	Mt	**0.8**	0.0	0.0	0.3	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Blast Assisted Cave	Mt	**8.6**	0.0	0.0	0.2	2.0	2.8	2.8	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Natural Cave	Mt	**19.1**							2.0	2.9	2.9	2.9	2.9	2.9	2.9	0.0
Total Draw	Mt	**28.5**	0.0	0.0	0.5	2.4	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	0.0
Hoisted Material																
Hoisted Ore	Mt	**28.5**	0.0	0.0	0.5	2.4	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	0.0
EM/PK(S)	Mt	**14.2**	0.0	0.0	0.5	2.3	2.5	2.2	1.4	0.9	0.6	0.6	0.8	1.0	1.3	0.0
M/PK(S)	Mt	**14.3**	0.0	0.0	0.0	0.2	0.3	0.6	1.4	2.0	2.2	2.2	2.1	1.8	1.5	0.0
Hoisted Grade	cpht	**13.2**	0.0	15.0	15.1	18.2	18.5	18.2	15.7	13.6	12.7	12.1	10.4	7.7	5.7	0.0
EM/PK(S)	cpht	**17.1**	0.0	15.9	15.4	17.9	18.5	19.7	20.5	20.4	20.3	18.9	15.4	10.9	7.6	0.0
M/PK(S)	cpht	**8.6**	0.0	0.1	6.5	8.8	9.6	10.2	10.5	10.5	10.4	9.6	8.0	5.7	4.0	0.0
Hoisted Carats	kc	**3,773**	0	2	78	445	516	507	447	389	361	345	298	221	163	0
EM/PK(S)	kc	**2,429**	0	2	75	411	460	431	296	175	125	122	118	114	101	0
M/PK(S)	kc	**1,240**	0	0	2	13	29	62	148	211	234	213	166	104	60	0
Hoisted Waste	Mt	**0.9**	0.1	0.2	0.5	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Hoisted	Mt	**29.4**	0.1	0.2	1.0	2.6	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	0.0

Notes:
1. Forward looking from October 2025.

16.12.4 Combined OP and UG Production Schedule

The OP and UG mine production schedule for KDM incorporates South Lobe reserves mined from both OP and UG operations as well as North, Centre, and South Lobe reserves mined from stockpile. The OP will operate until April 2026; with existing surface stockpiles being consumed as processing capacity comes available. After completion of the OP, mill feed will be provided from the existing stockpiles, until the UGP can fully sustain mill feed requirements in late 2028. After completion of the UGP at the end of 2037, the remainder of the existing stockpiles will be processed. The OP production targets and mine plan was prepared by the Lucara Botswana Mine Technical Services team and verified by the QP.

The mill blending and stockpiling strategy after the completion of the OP in April 2026 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 16-37 summarises the combined LOM production schedule for KDM, including the OP and UG mines, the mill feed schedule, and stockpile balances. Note that 2025 values are a combination of actuals from Q1 to Q3 2025 and forecasts from Q4 2025.

Table 16-37: Combined LOM Production Schedule

Description	Unit	Total	2025 [1]	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Mining Summary																
Waste - OP Mining	Mt	**0.0**	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore - OP Mining	Mt	**1.1**	0.5	0.6	-	-	-	-	-	-	-	-	-	-	-	-
Ore - UG Mining	Mt	**28.5**	-	0.0	0.5	2.4	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	-
Mill Feed																
Direct Feed	Mt	**29.6**	0.5	0.6	0.5	2.4	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	-
From Stockpiles	Mt	**7.7**	0.2	2.2	2.3	0.4	0.1	0.1	-	-	-	-	-	-	-	2.4
	Mt	**37.3**	**0.7**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.9**	**2.4**
Total Mill Feed	000's ct	**4,547**	85.11	363	330	446	492	495	444	385	358	334	284	218	161	152
	cpht	**12.2**	12.3	12.7	11.6	15.6	17.3	17.4	15.6	13.5	12.6	11.7	10.0	7.6	5.6	6.4
Mill Feed - By Domain																
	Mt	**0.4**	-	-	-	-	-	-	-	-	-	-	-	-	-	0.4
North	000's ct	**51**	-	-	-	-	-	-	-	-	-	-	-	-	-	51
	cpht	**13.9**	-	-	-	-	-	-	-	-	-	-	-	-	-	13.9
	Mt	**2.9**	0.1	1.3	1.5	-	-	-	-	-	-	-	-	-	-	-
Centre	000's ct	**408**	15	202	191	-	-	-	-	-	-	-	-	-	-	-
	cpht	**14.2**	14.5	15.8	12.8	-	-	-	-	-	-	-	-	-	-	-
	Mt	**0.1**	0.1	0.0	-	-	-	-	-	-	-	-	-	-	-	-
OP-South-EM/PK(S)	000's ct	**29**	21	9	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	**21.2**	21.6	20.3	-	-	-	-	-	-	-	-	-	-	-	-
	Mt	**1.9**	0.4	1.5	-	-	-	-	-	-	-	-	-	-	-	-
OP-South-M/PK(S)	000's ct	**192**	42	150	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	**10.1**	10.8	9.9	-	-	-	-	-	-	-	-	-	-	-	-
	Mt	**3.5**	0.1	-	0.8	0.4	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.0
LOM	000's ct	**198**	8	-	62	21	3	3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	101
	cpht	**5.7**	7.4	-	7.4	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1
	Mt	**14.2**	-	0.0	0.5	2.3	2.5	2.2	1.4	0.9	0.6	0.6	0.8	1.0	1.3	-
UG-South-EM/PK(S).	000's ct	**2,429**	-	2	75	411	460	431	296	175	125	122	118	114	101	-
	cpht	**17.1**	-	15.9	15.3	17.9	18.5	19.7	20.5	20.4	20.3	18.9	15.4	10.9	7.6	-
	Mt	**14.3**	-	0.0	0.0	0.1	0.3	0.6	1.4	2.0	2.2	2.2	2.1	1.8	1.5	-
UG-South-M/PK(S)	000's ct	**1,240**	-		2	13	29	62	148	211	234	213	166	104	60	-
	cpht	**8.6**	-	0.1	7.1	9.0	9.6	10.2	10.5	10.5	10.4	9.6	8.0	5.7	4.0	-
Stockpile Inventory																
North																
High Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Medium Grade	Mt		0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4

Description	Unit	Total	2025 [1]	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Low Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Centre																
High Grade	Mt		1.0	0.9	-	-	-	-	-	-	-	-	-	-	-	-
Medium Grade	Mt		1.9	1.8	1.5	-	-	-	-	-	-	-	-	-	-	-
Low Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	
OP - South-EM/PK(S)																
High Grade	Mt		0.0	0.0	-	-	-	-	-	-	-	-	-	-	-	
Medium Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	
Low Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	
OP - South-M/PK(S)																
High Grade	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	
Medium Grade	Mt		0.3	0.3	-	-	-	-	-	-	-	-	-	-	-	
Low Grade	Mt		0.6	0.6	-	-	-	-	-	-	-	-	-	-	-	
UG																
EM/PK(S) + M/PK(S)	Mt		-	-	-	-	-	-	-	-	-	-	-	-	-	
Life of Mine																
Contact	Mt		0.7	0.6	0.6	-	-	-	-	-	-	-	-	-	-	-
LOM	Mt		2.8	2.7	2.7	2.5	2.1	2.1	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Total Stockpile	**Mt**		**7.6**	**7.5**	**5.2**	**2.9**	**2.5**	**2.4**	**2.4**	**2.4**	**2.4**	**2.4**	**2.4**	**2.4**	**2.4**	**2.4**

Notes:
1. Forward looking from October 2025.

The total blended mine and mill feed from both UG, OP, and stockpile operations is shown in Figure 16-98 and Figure 16-99.

Figure 16-98: Summary of Mine Production

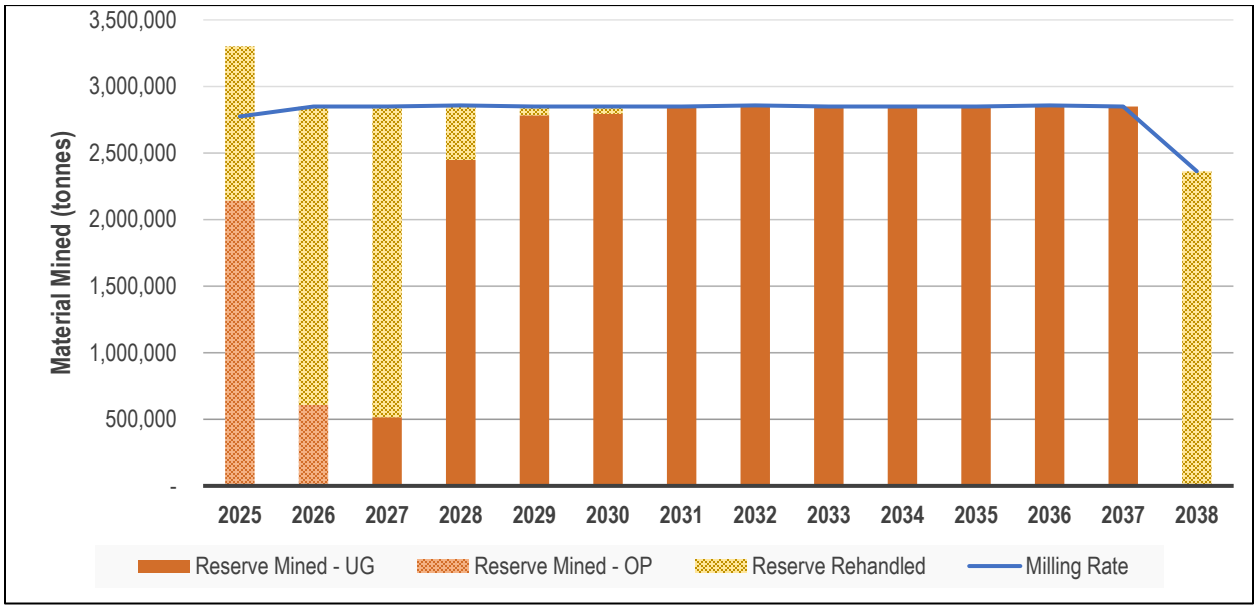
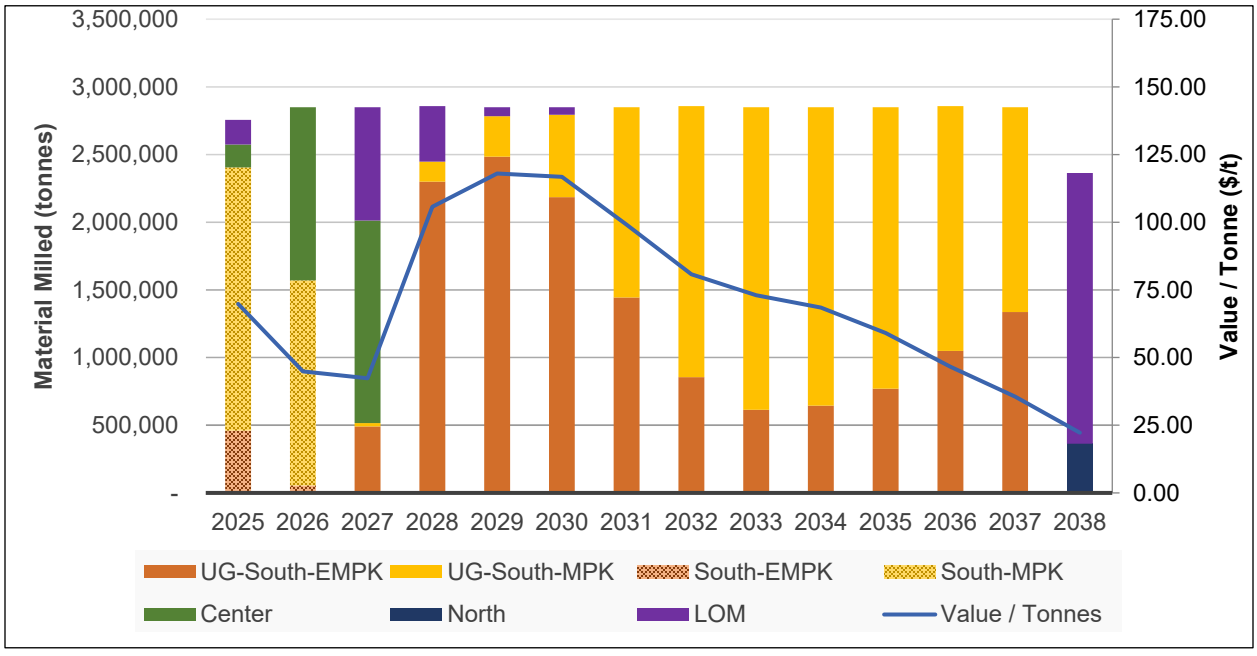
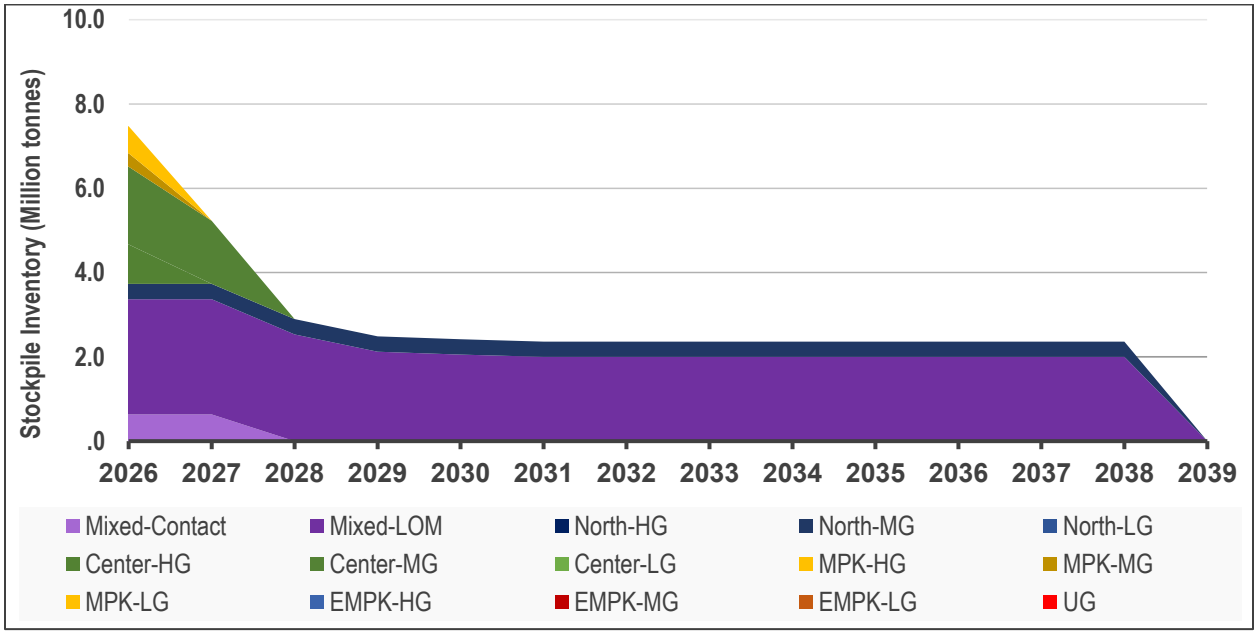


Figure 16-99: Summary of Mill Production



A summary of the stockpile inventory opening balance is outlined in Figure 16-100.

Figure 16-100: Summary of Stockpile Inventory Opening Balance



17 PROCESS DESCRIPTION / RECOVERY METHODS

17.1 Introduction

The QP, DRA Projects Pty Ltd. (DRA) was commissioned by Lucara Diamond Corp. (Lucara) to perform an overall treatment plant evaluation 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; and
- Tiroyaone Kesiilwe, Lucara Botswana Senior Process Engineer – Technical.

The following subsections 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 and Dense Media Separation (DMS), recovery plant, associated crushing, screening and thickener systems for KDM (called AK06 Mine at that time).

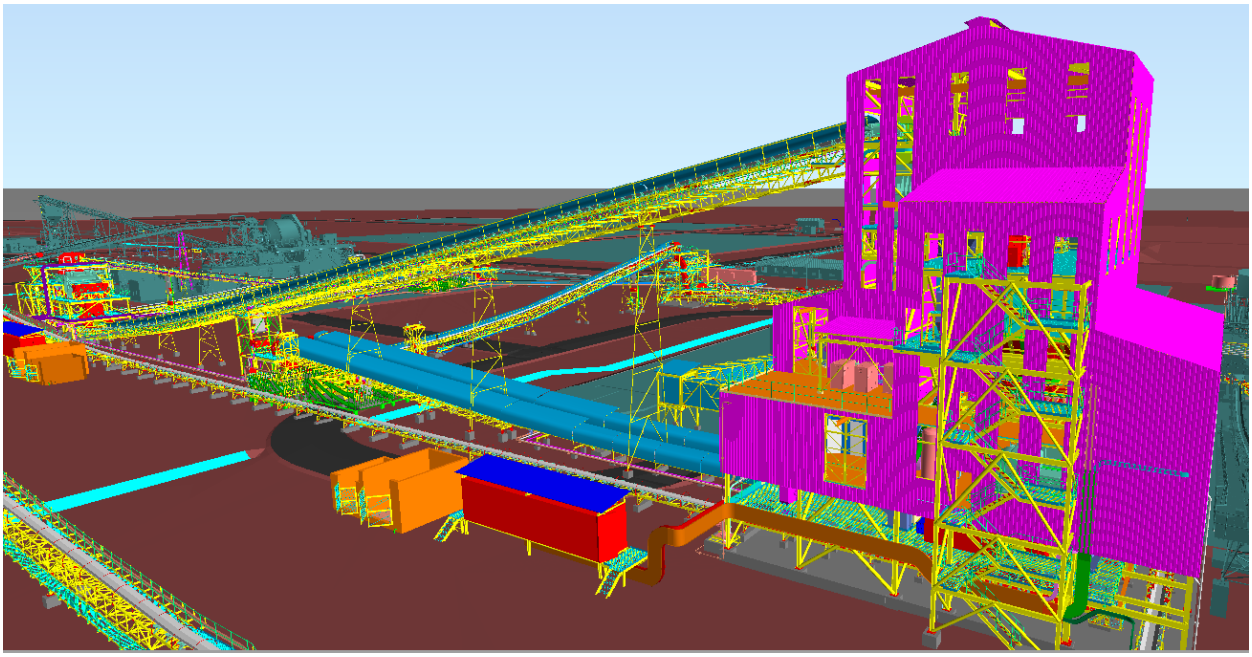
KDM was designed to process 2.5 Mt of Run of Mine (ROM) kimberlite ore per annum with a single 200 t/h DMS module. The concentrate material from the DMS was subsequently treated through a 2.5 t/h wet X-ray Recovery for material reduction and diamond winning. Adequate space was allowed for during the 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 utilised 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 recognise 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 utilised 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)

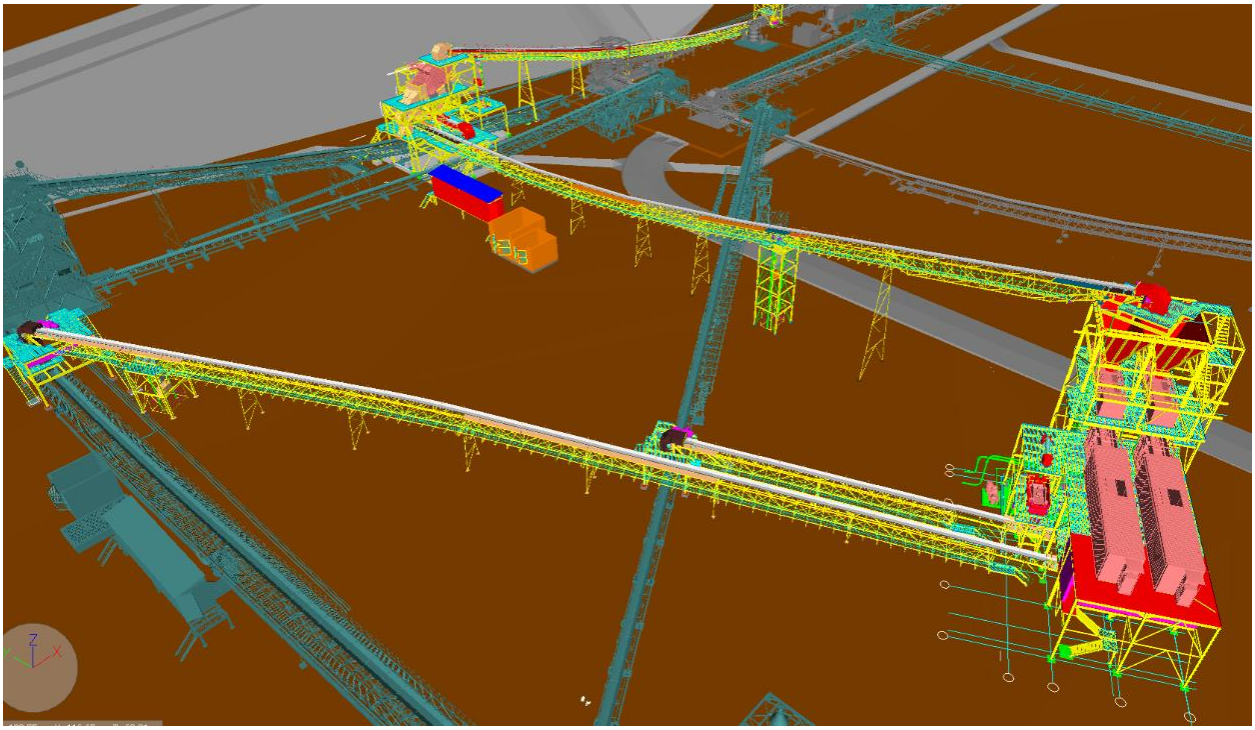
17.1.3 KDM Mega Diamond Recovery and Phase III (Brownfields) History

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,

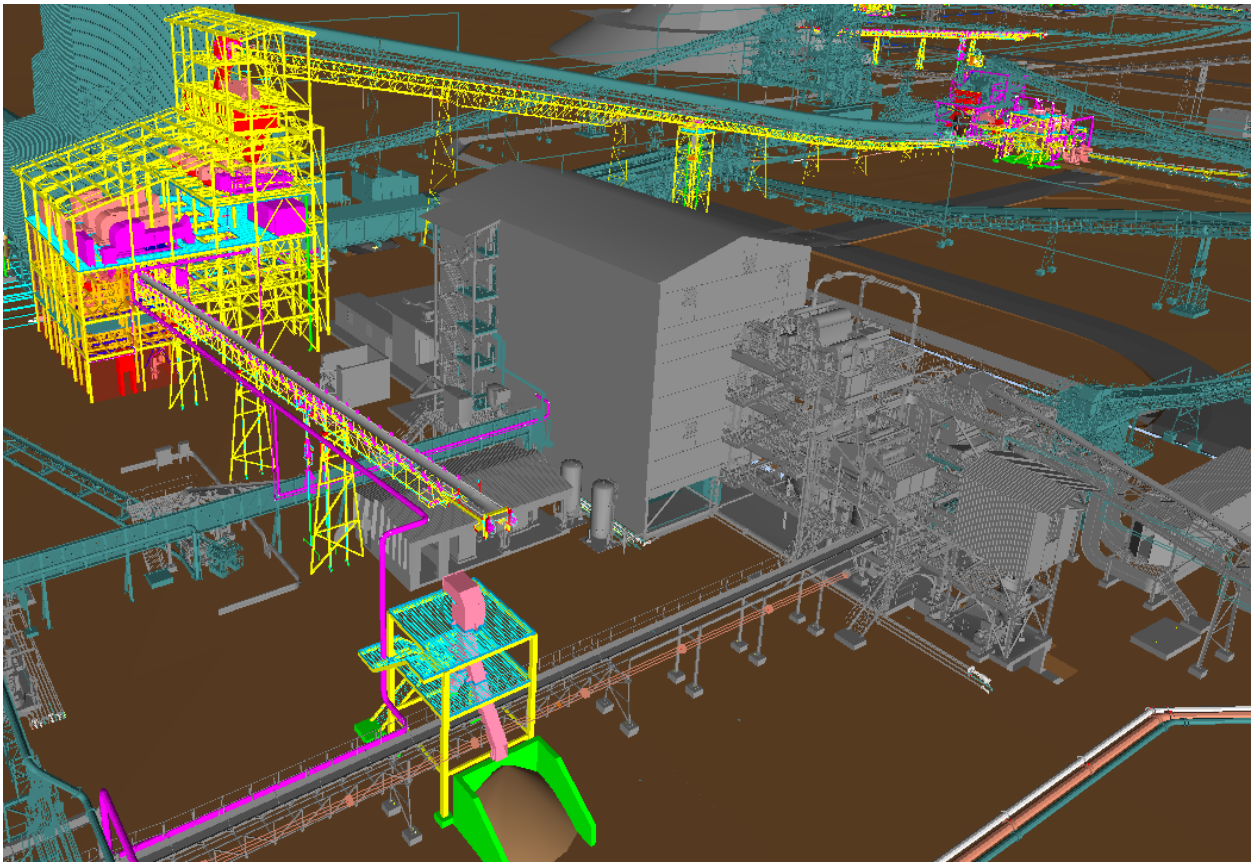
construction, and commissioning of the Phase III brownfields expansion. Phase III made provision for the inclusion and application of XRT sorting technology to the 4 x 8 mm size fraction ahead of the DMS – with the ultimate aim of negating the high-frequency near density content of KDM's Unit 13 (M/PK(S)) 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.

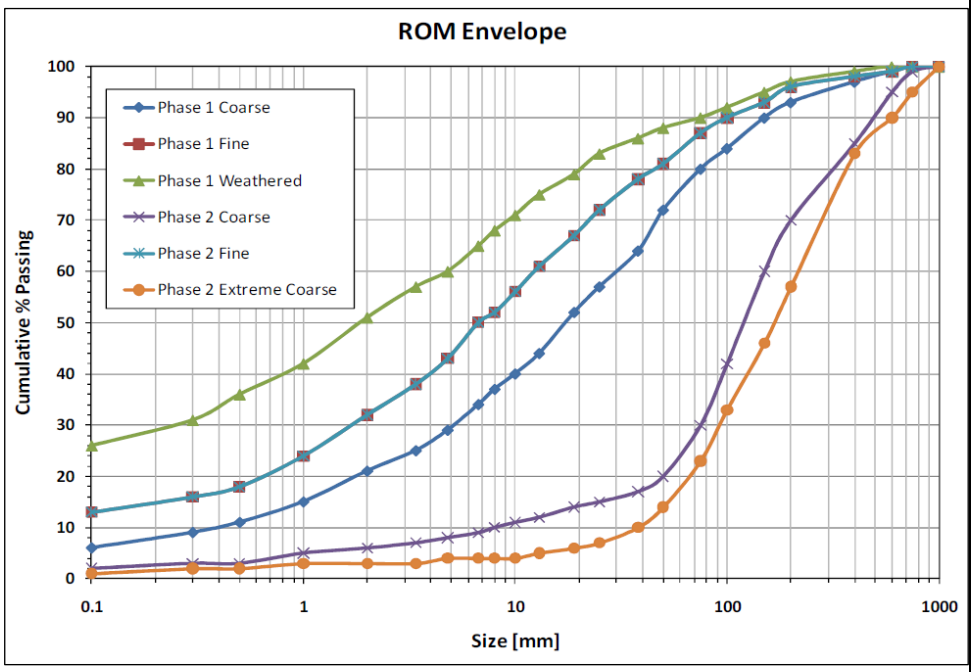
The following source codes are used to reference the origin of each item of information that appears in the design criteria.

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

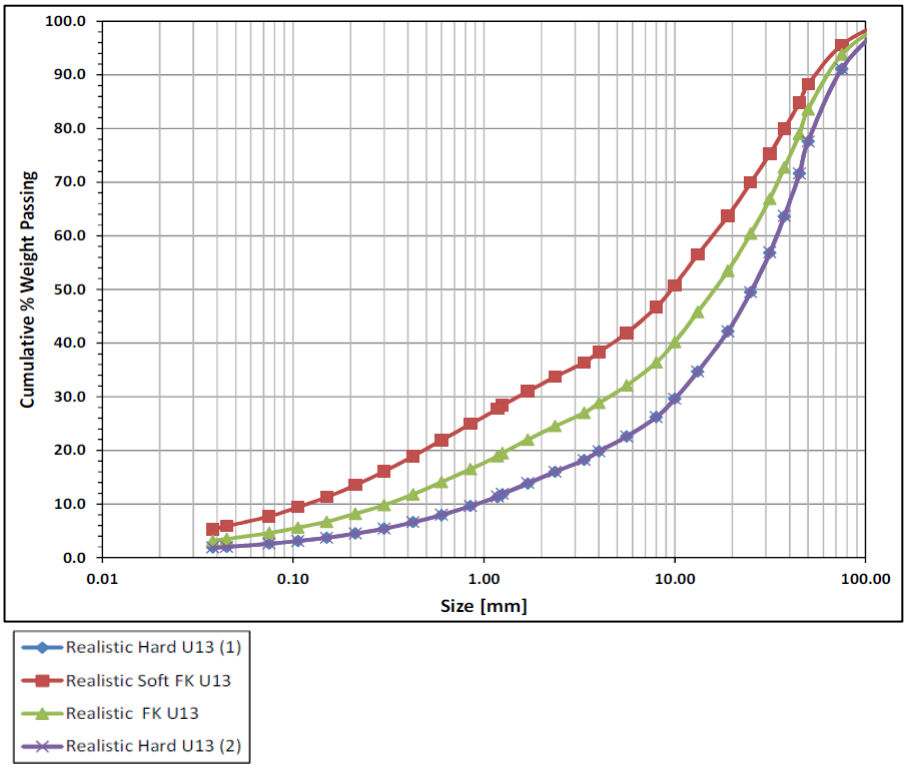
Source: DRA (2014)

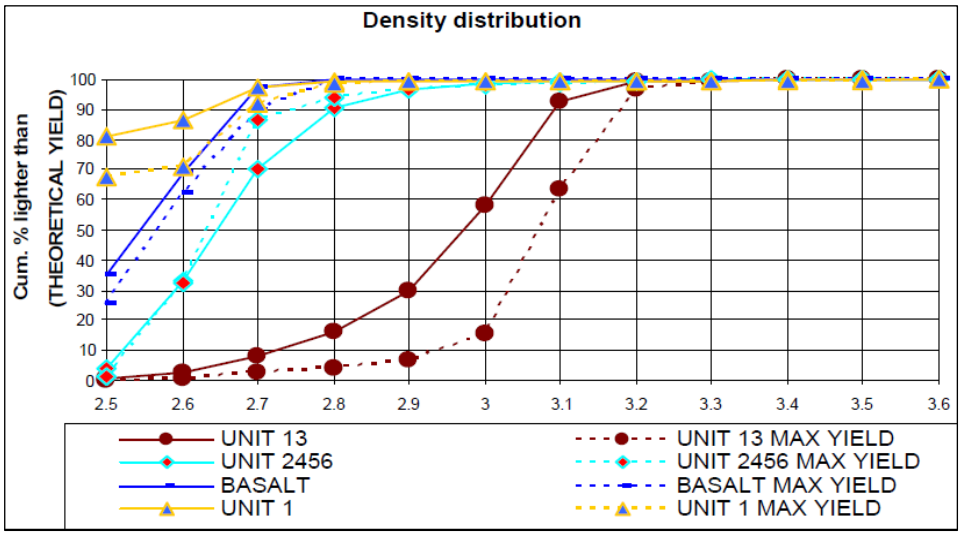
Table 17-2: Process Design Criteria

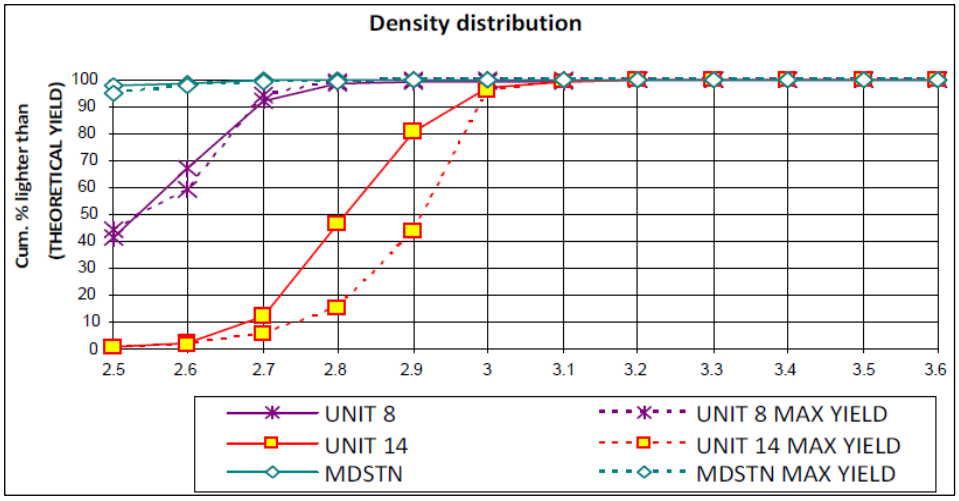
Criteria	Units	Value	Source	Revision
Ore type to be treated	-	Diamond bearing kimberlite	C	A
Design annual tonnage	dry mt/a	2.5 - 3.5	C	0
Manned hours per annum	hrs pa	8,760	C	A
Overall utilisation	%	81.0	D3	A
"On ore" hours per year	hrs pa	7,095	D3	A
Design throughput	t/h	350 - 500	C	A
Operation type	-	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

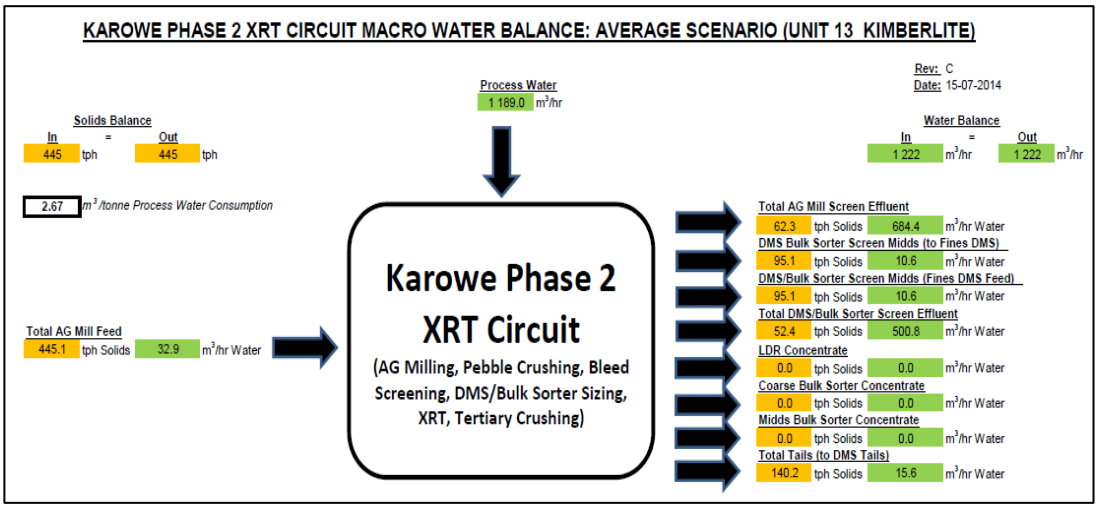
Note: "Phase 2" in graph above denotes unweathered M/PK(S) kimberlite ore type

Criteria		Units	Value	Source	Revision
Crushability Data					
Description		**T10 [%] @ 0.25 kWh/t**	**T10 [%] @ 1 kWh/t**		
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		
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 – 75 mm	D3	A
AG Milling					
Discharge grate		-	TPL type grate	D2	A
Circuit Feed Size (Fresh Feed): F_{80}		mm	~125.0	D2	A
Circuit Product Size: P_{80}		mm	~37.5 - 50.0	D2	A
Circuit Product Size: % -1.5 mm		%	~13 - 30	D1	A

Criteria	Units	Value	Source	Revision
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 (-1.5 mm, before dilution)	t/m^3	1.09	D1	A
Product Slurry Density Target (-1.5 mm, 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
-32 mm mill bypass	%	0, 12.5, 25, 37.5, 50, 62.5, 75, 100	D3	A
XRT Bulk Sorters				
Technology	-	XRT	D2	A

Criteria	Units	Value	Source	Revision
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
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
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				
-8 +4 mm	%	60.0	D2	A
-4 +1.5 mm	%	40.0	D2	A
Average Yield Throughput				
+4 mm	t/h	5.88	D1	A
+1.5 mm	t/h	3.92	D1	A
75 Percentile Throughput				
+4 mm - Middles	t/h	9.24	D1	A
+1.5 mm - Fines	t/h	6.16	D1	A
Wet MagRoll Capacity (Based on 2 Streams)				
+4 mm - Middles 5 t/h	t/h	10.0	V	A
+1.5 mm - Fines 3 t/h	t/h	6.00	V	A
MagRoll Reduction	%	65.0	D2	A
Wet X-ray Capacity (Based on 2 Streams)				
+4 mm - Middles 1950 kgh	t/h	4.00	V	A
+1.5 mm - Fines 1050 kgh	t/h	2.00	V	A
Feed to X-ray Circuit				
Average Yield +4 mm - Middles	t/h	2.06	D1	A
Average Yield +1.5 mm - Fines	t/h	1.37	D1	A

Criteria	Units	Value	Source	Revision
75 Percentile Yield +4 mm - Middles	t/h	3.23	D1	A
75 Percentile Yield +1.5 mm - Fines	t/h	2.16	D1	A
Reconcentration X-ray Capacity				
+4 mm - Middles	kgh	25.0	V	A
+1.5 mm - Fines	kgh	10.0	V	A
Phase 2 Macro Water Balance				
				0

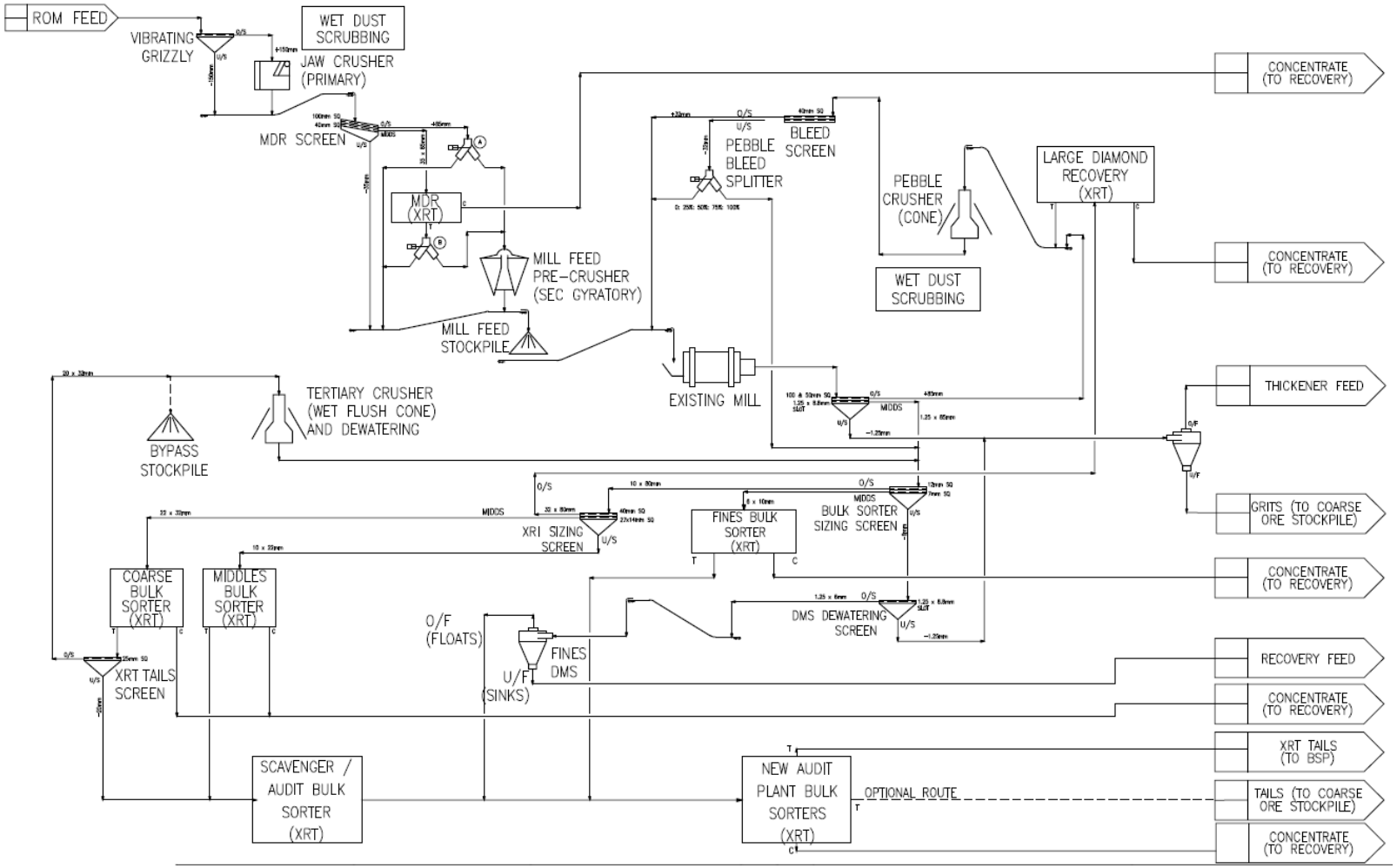
Source: DRA (2014)

17.3 Plant Design and Current Plant Performance

Figure 17-5 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 denote the original equipment from Phase I, whilst the equipment and streams highlighted in green font denote subsequent changes post the Greenfields Phase I build. 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.

Figure 17-5: Overall KDM Block Flow Diagram (Current)



Source: DRA (2025)

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 4.00 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 2019 site visit as well as the 15 October 2024 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 utilised 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;

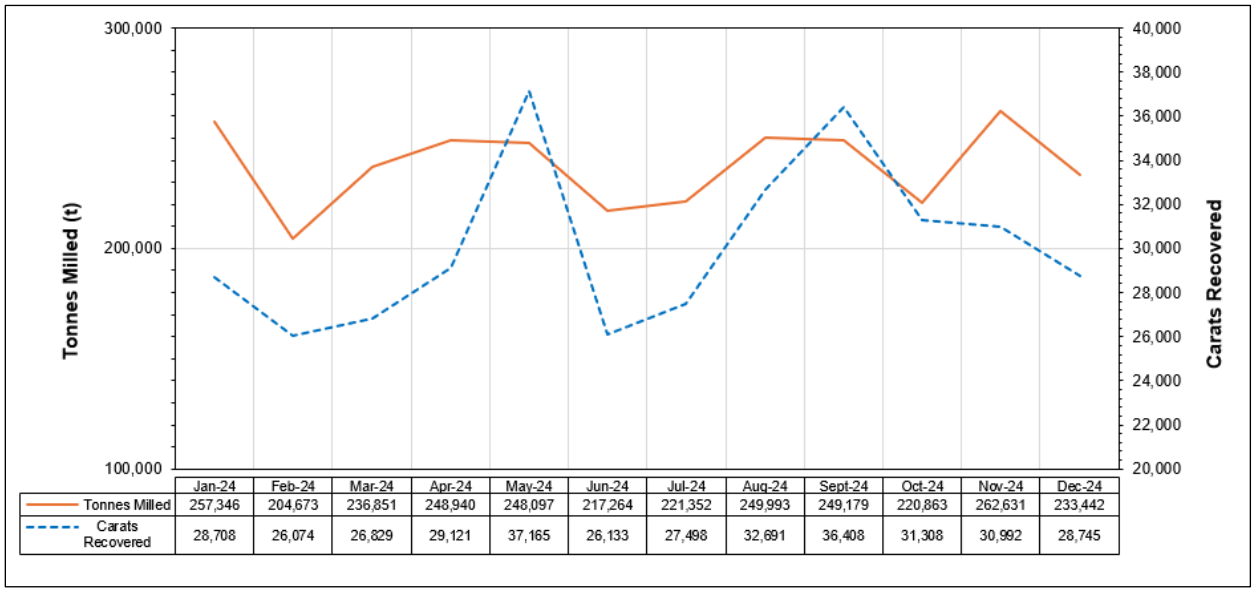
- XRT Sorthouse upgrade was completed on 03 December 2018. Holding bins, feeders, washer driers and sort boxes were installed as part of the overall project. The main aim of the XRT Sorthouse upgrade project was to improve on washing and drying the concentrate product for increased (manual sorting) visibility. 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;

- A second Coarse Disposal tailings leg was introduced to assist with coarse tails deposition and storage. This change has also assisted with operational flexibility in terms of plant utilisation and availability; and

- A new Tailings Storage Facility (TSF, or Slimes Dam) was introduced at KDM for slimes collection and water recovery/reuse back in the processing plant.

The following two graphs summarise 2024 plant performance in terms of milled tonnage, carats recovered and key treatment plant feed stream Particle Size Distribution (PSD) data.

Figure 17-6: 2024 Crushed/Milled Tonnage vs. Carat Recovery

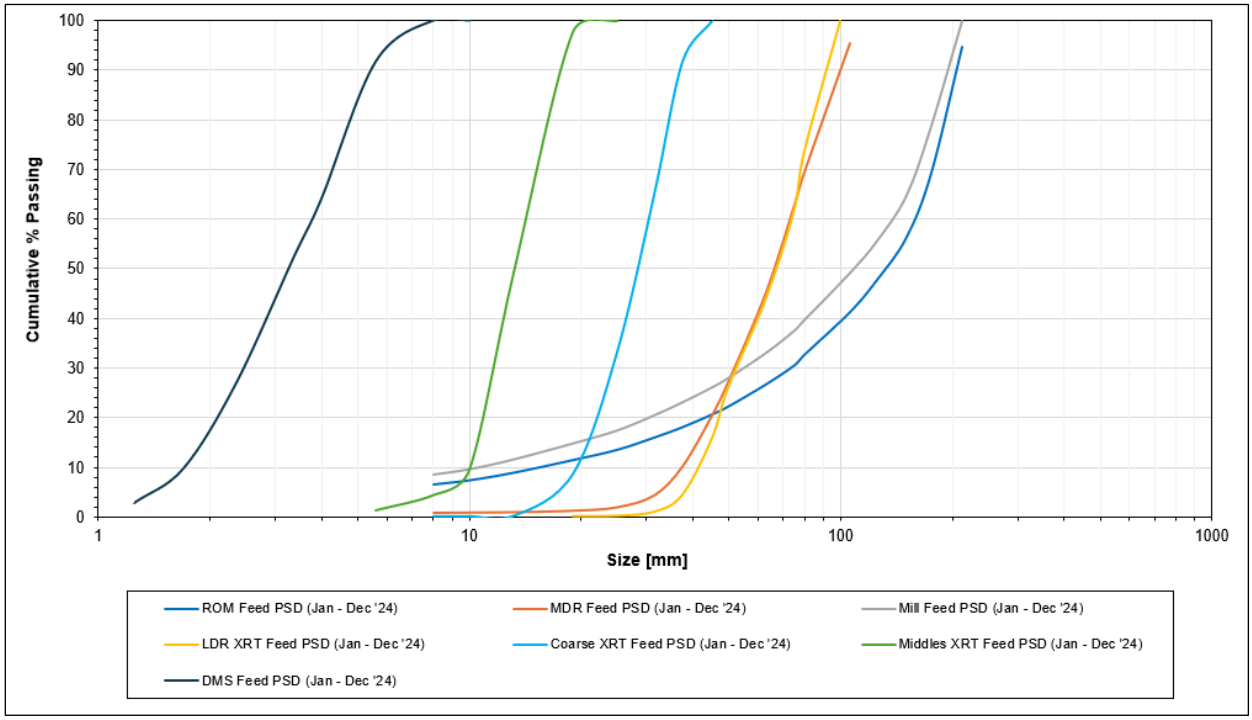


	Jan-24	Feb-24	Mar-24	Apr-24	May-24	Jun-24	Jul-24	Aug-24	Sept-24	Oct-24	Nov-24	Dec-24
Tonnes Milled	257,346	204,673	236,851	248,940	248,097	217,264	221,352	249,993	249,179	220,863	262,631	233,442
Carats Recovered	28,708	26,074	26,829	29,121	37,165	26,133	27,498	32,691	36,408	31,308	30,992	28,745

Source: Lucara (2025)

From above graph, slightly lower monthly production was observed for February, June, July, October and December 2024 months respectively. The decreased production during these months was attributed to either one of the following events: planned shutdowns, unexpected equipment breakdowns and/or failures, and power outage(s).

Figure 17-7: 2024 Treatment Plant Key Feed Stream PSDs



Source: Lucara (2025)

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 Sizing Screen (Double Deck)	Top: 125 x 100 mm SLOT Bottom: 40 mm SQ
Bleed Screen (Double Deck)	Top: 40 mm SQ Bottom: Solid/Blank
Mill Discharge Screen (Double Deck)	Top: 100 mm and 50 mm SQ (combination) 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

Screen Description	Screen Panel Aperture Size
XRT Tails Screen (Single Deck)	25 mm SQ
DMS Dewatering Screen (Single Deck)	1.25 x 8.8 mm SLOT

Notes:
SQ denotes square aperture and SLOT denotes slotted aperture.
Source: Lucara (2025)

Table 17-5: Crusher CSS Summary

Crusher Description	Closed Side Setting (CSS) Size (mm)
Primary Jaw Crusher	165 – 180
Secondary Gyratory Crusher	65 – 85
Pebble Crusher	28 – 35
Tertiary Wet Flush Crusher	13.5 – 15.0

Source: Lucara (2025)

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 3 million tonnes per annum (Mtpa). The following key points summarises 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; and

- 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 and Bleed Screening

Ore from mining is tipped into the Primary Crusher Feed Bin. This is equipped with a Static Grizzly on top to ensure only -800 mm material reports into the feed bin. Oversize material is broken down with the help of an in situ rock breaker. From the bin, the material is withdrawn at a controlled rate by an Apron Feeder (with variable speed drive), to regulate the feed into the Jaw Crusher chamber. Based on the feed rate and crusher chamber cavity level setpoint controls, the Apron Feeder material is discharged onto the Vibrating Grizzly (with 150 mm gap), which screens out material into two products. The +150 mm oversize material reports to the Jaw Crusher which crushes material down to 180 mm; and the -150 mm undersize material bypasses the Jaw Crusher and combines with the Jaw Crusher product. This crusher operates at 550 tonnes per hour (tph) maximum.

The Primary Crusher is a single toggle jaw crusher. Material is fed from the top into the crusher and is crushed between two jaws – the fixed and the swing (moveable) jaw. The product size is determined by the jaw gap setting which is set at between 165-180 mm. The combined material from the primary section is then conveyed to a sizing screen, called the MDR Sizing Screen. The main purpose of the screen is to separate out the target size range to be fed to the Mega Diamond Recovery (MDR) section. The top deck of the MDR Screen has 125 mm apertures, with a design cut-point of 100 mm; while the bottom deck has 40 mm apertures with a design cut-point of 32 mm. From this screen, three products are generated: undersize product (-32 mm), middlings product (+32 -100 mm), and an oversize product (+100 mm -180 mm). Undersize product is classified as a fine fraction, middlings product reports to the MDR section, and oversize product reports to the Secondary Bin.

If required, a portion of the material is transferred to the Secondary Crusher for size reduction; otherwise, it's conveyed to the Mill Feed Stockpile. The decision to send material to the Secondary Crusher depends on the amount of oversized rock (i.e., how coarse the feed is). Only a portion of Secondary Bin material is sent to the Secondary Crusher (if necessary / required); otherwise, all material bypasses the crusher to the Mill Feed Stockpile.

The main purpose of the Secondary Crusher circuit is to ensure that feed that ultimately reports to the mill is of desired (or targeted) size distribution as this can affect AG Mill grinding performance. The middlings product is fed to the MDR section to recover any liberated diamonds through the XRT machines. The MDR section operates on two XRT streams, each with a maximum throughput of 150 tph. The undersize product combines with MDR tailings (MDR treated material), Secondary Crusher bypass and secondary product material – and this is sent

to the Mill Feed Stockpile. All feed from the ROM section ultimately reports to the Mill Feed Stockpile, which has a holding (buffer) capacity of about 24,000 tonnes (3 days feed). The ROM front-end section (with exception of MDR) is a dry plant.

17.4.2 Comminution – Milling and Pebble Crushing

Karowe Diamond Mine operates an autogenous grinding mill (AG Mill). From the Mill Feed Stockpile, feed is withdrawn at a controlled rate from the bottom of the stockpile using two apron feeders (with variable speed drives) and conveyed to the AG Mill. The mill on average operates at 400 tph of fresh feed (i.e., feed from stockpile). Design throughput is close to 450 tph of fresh feed. The AG Mill utilises the rocks as grinding media for size reduction – hence the feed size range reporting to the mill is crucial in achieving the desired or targeted product grind. Material is fed into the mill which rotates and lifts the material from the toe by the aid of lifter bars. Water is added with feed to the mill to assist in movement of charge (feed within the mill) and the load of the mill is maintained to approximately 240 tonnes.

The AG Mill is equipped with Turbo Pulp Lifters (TPLs) for easier discharge of product from the mill. It also utilises discharge grates (on the inside) with various port (openings) sizes – the largest port size being 115 mm, to ensure that only -115 mm material exits the mill. Mill product reports to the Mill Discharge Screen, which is a double deck screen for washing, draining, and sizing. Top deck of the mill discharge screen has 100 mm apertures and achieves a cut-point of 90 mm. The bottom deck removes fine slurry generated by the mill and screens out the 1.25 mm material (which is the bottom cut-off size for KDM). From this screen, three products are generated with the following size ranges: undersize product (-1.25 mm), middlings product (+1.25 – 90 mm), and oversize product (+90 mm). Undersize product reports to the Degrit and Thickening section for slimes disposal and water recovery. Middlings product is transferred to the Bulk Sorter Sizing Screen and oversize product reports to the Pebble Crusher for re-crushing (material size reduction).

The Pebble Crusher is a cone crusher that operates in a closed circuit with the AG Mill. Material is introduced from the top of the crusher and is crushed between the fixed crushing surface called the concave and the moveable crushing surface called the mantle (which moves in a gyratory motion within the concave cone). The smallest product size is determined by the closed side setting (CSS) of the crusher which is set at a gap between 28 to 35 mm – and the material is compressed and discharged at a desired product range of between 35 to 42 mm. Crusher product is screened at the Bleed Screen, and the oversize returned back to the AG Mill as recycle.

17.4.3 Bulk Sorter Sizing and XRT

Mill Discharge Screen middlings product is sent to the Bulk Sorter Sizing Screen for additional sizing purposes to prepare feed going to various downstream sections. The Bulk Sorter Sizing Screen is a double deck wash screen. Top deck has 12 mm apertures cutting at 9.6 mm, whilst the bottom deck has 7 mm apertures cutting at 5.6 mm. From this screen, three products are generated: an undersize product (-5.6 mm) which is labelled DMS feed size material with effluent. This material is pumped to a dewatering screen to dewater the product and generate DMS feed material. Effluent from the dewatering screen is pumped to the thickener for water recovery. DMS feed material (post dewatering) is conveyed to the DMS Feed Bin. Bulk Sorter Sizing Screen middlings product (+5.6 -9.6 mm) is associated with 4-8 mm XRT feed size material which is

conveyed to the 4-8 mm XRT section for treatment. Bulk Sorter Sizing Screen oversize (+9.6 -90 mm) material is reserved for main XRT feed size material which is conveyed to the main XRT section for further sizing (or size separation), followed by treatment.

Bulk Sorter Screen oversize product is conveyed to the Main XRT section. At the Main XRT section, feed is classified into three size fractions via a double deck sizing screen (that operates dry). Top deck has 40 mm apertures, cutting at 32 mm and the bottom deck has 14 mm slotted (14 x 27 mm) apertures cutting at 17 mm (actual cut off design is 14 mm). From this screen, three products are generated with the following size ranges: undersize product, -17 mm (+9.6 -17 mm) which is the feed to the Middles XRT stream. A middlings product stream is also generated (+17 -32 mm), which becomes feed to the Coarse XRT stream. Oversize Product, +32 mm (+32 -90 mm) is feed to the LDR stream. After sizing, material is fed into the respective feed bins for each size fraction (Middles, Coarse and LDR). Middles and Coarse streams operate with identical two-stream configurations, while the LDR operates only with one stream configuration. Middles, Coarse and LDR have a maximum throughput of 60 tph (120 tph total), 100 tph (200 tph total) and 150 tph respectively. For each stream, material is treated through the respective XRT machines and the concentrate generated by each of the five machines is sent to the Sorthouse for hand sorting.

Tailings from both Middles streams is conveyed to the coarse tailings dump and forms part of final exit tailings. Tailings from the Coarse streams is subjected to screening at 20 mm: where the +20 mm material is conveyed to the tertiary crusher for further size reduction. The working principle of the tertiary crusher is the same as the pebble crusher as they are both cone crushers, however the tertiary utilises wet flush crushing and has a closed side setting of between 13 – 15 mm and the desired product size is between a range of 15 – 20 mm. After crushing, the product is conveyed to the Bulk Sorter. Material that is smaller than 20 mm is conveyed to the coarse tailings dump and forms part of final exit tailings. Tailings from the LDR is conveyed to the Pebble Crusher for further size reduction. The Pebble Crusher operates in a closed circuit with the AG Mill. Pebble Crusher product oversize returns to the AG Mill, while pebble product undersize reports to the Bulk Sorter Sizing Screen.

Bulk Sorter Screen Middlings product (+5.6 -9.6 mm) is conveyed to the 4-8 XRT section. This section was originally designed to treat +4 -8 mm material hence the name ‘4-8 XRT’. The material is split into three identical streams and is stored in the respective feed bins. From each bin, material is withdrawn and processed through the respective XRT machine for diamond recovery. This is a similar process to the Main XRT section but targeting a smaller size range. The maximum throughput for each machine is 25 tph, bringing the (collective) total maximum throughput to 75 tph. Concentrate from each of the three 4-8 machines is reconcentrated through another XRT machine (recon machine) to reduce yield for hand sorting. Tailings from the recon machine is deposited to the red area tailings. Tailings from first stage of (primary) treatment, is conveyed to the coarse tailings dump and forms part of final exit tailings.

17.4.4 DMS and Recovery

Dewatered oversize product (+1.25 -5.6 mm), from the Bulk Sorter circuit (Bulk Sorter undersize) is conveyed to the Dense Media Separation (DMS) section for gravity separation/beneficiation treatment. DMS utilises media made of a mixture of ferrosilicon (FeSi) and water at an average operating medium density of 2.70 t/m^3 – to separate material (concentrate and gangue) according to density. At an average operating medium density of 2.70 t/m^3 the DMS achieves a cut-point of 3.15 t/m^3, generating two products. DMS sinks/concentrate (+3.15 t/tm^3) is processed further in

the Recovery building through DebTech X-Ray luminescent (XRL) machines to reduce yield prior to hand sorting.

The XRL machines' separating principle is based on luminescence intensity. Material is presented into the machines by way of a spread-out monolayer distribution through a feeder tray. The X-Ray tube is powered by a generator at high voltage to generate an X-Ray beam focusing on the stream of material. The sorting of diamonds is based on the luminescent property of a diamond – when the beam hits the surface of a diamond it refracts the beam and sparkles, and the PM tube is set an angle to the material's path to detect the light refracted and activate the ejector channel to eject material into the concentrate chamber. The concentrate is sent to the Sorthouse for hand sorting. Tailings from the XRLs forms recovery tailings and is stockpiled in the Red Area tailings yard, as it is of a potentially higher grade. DMS floats/tailings (-3.15 t/m^3) is conveyed to the Coarse Tailings Dump and forms part of final exit tailings.

17.4.5 DMS Residue and Effluent Disposal

The Coarse Residue Disposal Facility stores tailings generated from the DMS, XRTs and grits. Thus, feed size range of material reporting to tailings is +300 µm -20 mm. The disposal facility has been in use from the first commissioning of the plant in 2012.

All slurry (-1.25 mm) or effluent is first pumped into the Degrit Cyclones (hydrocyclones) for further size classification prior to thickening. This is to reduce solids load to the thickener. Degrit Cyclones cut at an average of 300 µm, generating two cyclone products, namely underflow and overflow. Degrit Cyclone overflow feed is introduced into the thickener with addition of flocculant to assist in settling of solids. As solids settle to the bottom, clear water (or supernatant) is generated at the top and overflows into the Process Water Tank. The Thickener generates two products: underflow (or slimes), which has a density of 1.40 t/m^3 and is pumped to Slimes Dam 2 (Fines Residue Disposal, or FRD) facility for storage and water recovery. Thickener overflow reports to the Process Water Tank for recycling and reuse purposes.

The FRD is made up Slimes Dam 2 (or tailings storage facility) divided into two paddocks. The dam walls are constructed of waste rock and are extended as the paddocks become full. A deposition strategy is followed to manage dam capacity, water collection and return to the processing plant. Slimes is deposited into one paddock (at a time) until it reaches the allowable maximum capacity before shifting deposition to another paddock. After deposition, solids continue to settle and water is decanted, recovered and pumped back to the process plant.

17.4.6 Audit Plants

Old Audit Plant: a minimum of 10% of tailings material from the XRT middles sorters and XRT coarse tailings screens undersize is diverted to the Old Audit Plant for auditing purposes. The Old Audit Plant feed material is first washed and fines removed via a Dabmar screen. The Old Audit Plant has a single XRT sorter machine for sorting of the diverted tailings from the main XRT section. The concentrate from the XRT sorter is collected in a dock-lock canister for shipment to the Sorthouse (for further hand sorting). The tailings from the sorter is conveyed to the Coarse Tailings Dump for storage.

New Audit Plant: material is fed to the New Audit Plant facility via the Emergency Dump By-Pass Transfer Tower. At the transfer tower, a three-legged distributor is used to either direct material to the New Audit Plant facility (preferred/default route), the existing #1 Grasshopper Conveyor or the existing Emergency Dump Bypass Conveyor respectively. The New Audit Plant facility will only divert a portion (minimum 20%) of the main plant tailings stream, with excess material bypassing onto #1 Grasshopper Conveyor. The incoming feed is sized on a double deck screen; with the top deck cutting at 8.0 mm and the bottom deck at 4.0 mm. The Primary Sizing Screen oversize fraction reports to the Coarse XRT Feed Bin, the middles fraction reports to the Middles XRT Feed Bin and the undersize to the Secondary Sizing/Dewatering Screen. Oversize from the Secondary Sizing Screen is diverted to either a bunker facility for treatment through the Bulk Sampling Plant (BSP), or onto the outgoing Audit Tailings Conveyor. Underflow from the Secondary Screen reports to the Effluent Sump, from where it is pumped to a dedicated dewatering cyclone. Underflow from the secondary screen reports to the effluent sump from where it is pumped to a dedicated dewatering cyclone.

The New Audit Plant facility consists of two 1.2 m wide Tomra XRT units, one coarse and one middles machine. The XRT Feed Bins are each designed with overflow facilities, providing flexibility to maximise utilisation of both XRT units as the incoming feed particle size distribution changes. Two variable speed drive (VSD) fitted feeders extract and feed the material to the XRT Screens from where the oversize reports to the XRT units and the fines and undersize to the Effluent Sump. The XRT units are used for first and second pass processing. During first pass processing, XRT concentrate is collected in a Concentrate Hopper, from where it is jet pumped (once a day) for the second pass to the XRT units via the XRT Screens. Second pass concentrate from the XRT units is diverted into secure dock locks, which are collected daily and transferred to the main plant Sorthouse.

The XRT tailings, together with the XRT Feed Bin overflows and Secondary Screen oversize, report to the Audit Plant Tailings Conveyor, from where it is conveyed to the Coarse Tailings Disposal facility.

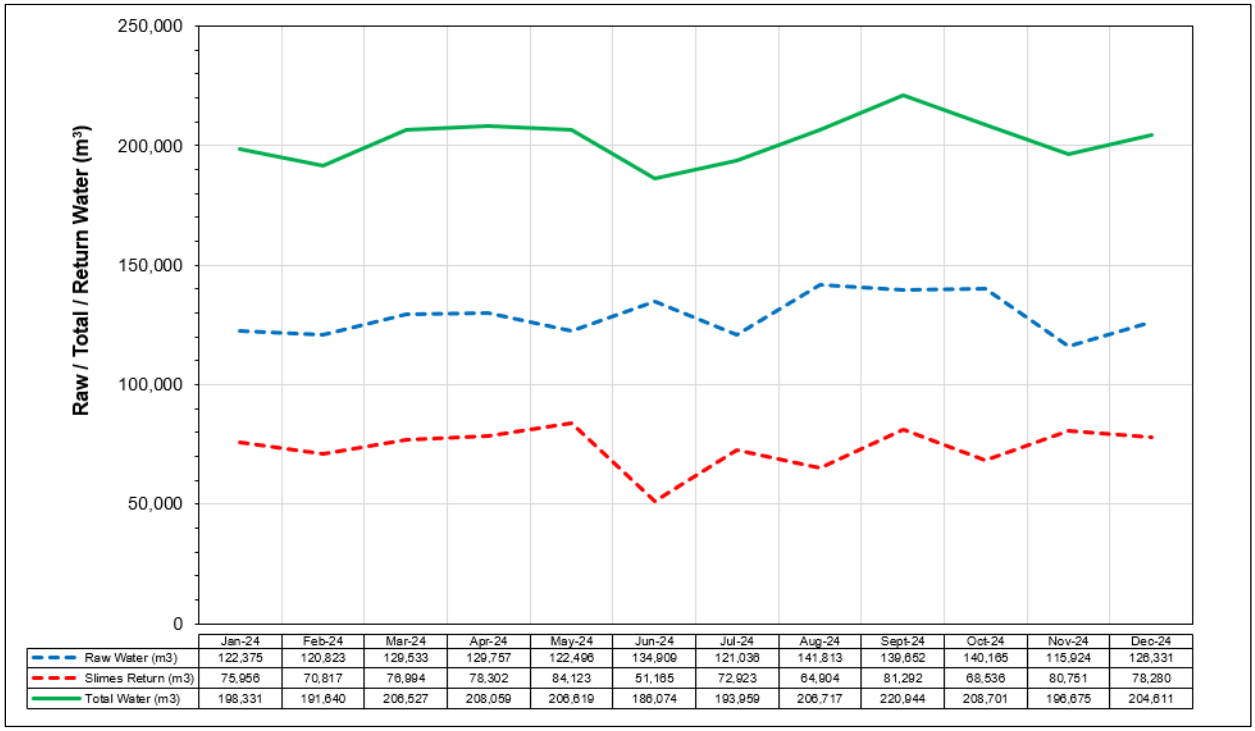
17.4.7 Services

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 expanded) R/O Plant. Water chillers in the XRT and Recovery sections continuously cool down equipment. Dust suppression is utilised 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 2024 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: 2024 KDM Raw/Total Water Consumption



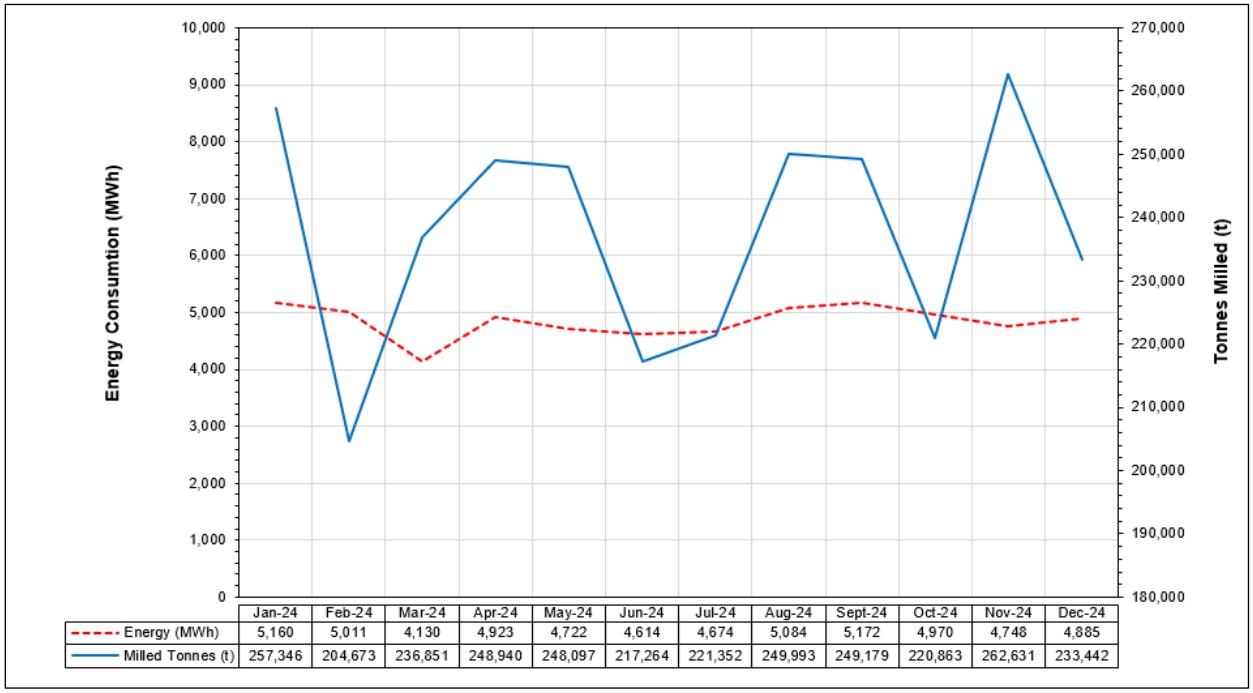
	Jan-24	Feb-24	Mar-24	Apr-24	May-24	Jun-24	Jul-24	Aug-24	Sept-24	Oct-24	Nov-24	Dec-24
Raw Water (m3)	122,375	120,823	129,533	129,757	122,496	134,909	121,036	141,813	139,652	140,165	115,924	126,331
Slimes Return (m3)	75,956	70,817	76,994	78,302	84,123	51,165	72,923	64,904	81,292	68,536	80,751	78,280
Total Water (m3)	198,331	191,640	206,527	208,059	206,619	186,074	193,959	206,717	220,944	208,701	196,675	204,611

Source: Lucara (2025)

17.4.9 Energy Consumption

Energy consumption data (associated with the process treatment plant) observed for the 2024 period is reported and summarised in Figure 17-9 below.

Figure 17-9: 2024 KDM Energy Consumption



	Jan-24	Feb-24	Mar-24	Apr-24	May-24	Jun-24	Jul-24	Aug-24	Sept-24	Oct-24	Nov-24	Dec-24
Energy (MWh)	5,160	5,011	4,130	4,923	4,722	4,614	4,674	5,084	5,172	4,970	4,748	4,885
Milled Tonnes (t)	257,346	204,673	236,851	248,940	248,097	217,264	221,352	249,993	249,179	220,863	262,631	233,442

Source: Lucara (2025)

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 leverages 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;
- Water Treatment Plant;
- Ancillary mobile equipment;
- Bulk fuel storage; and
- Offices, Security, Warehouses, and other miscellaneous facilities.

Since 2020 The UGP has added several 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

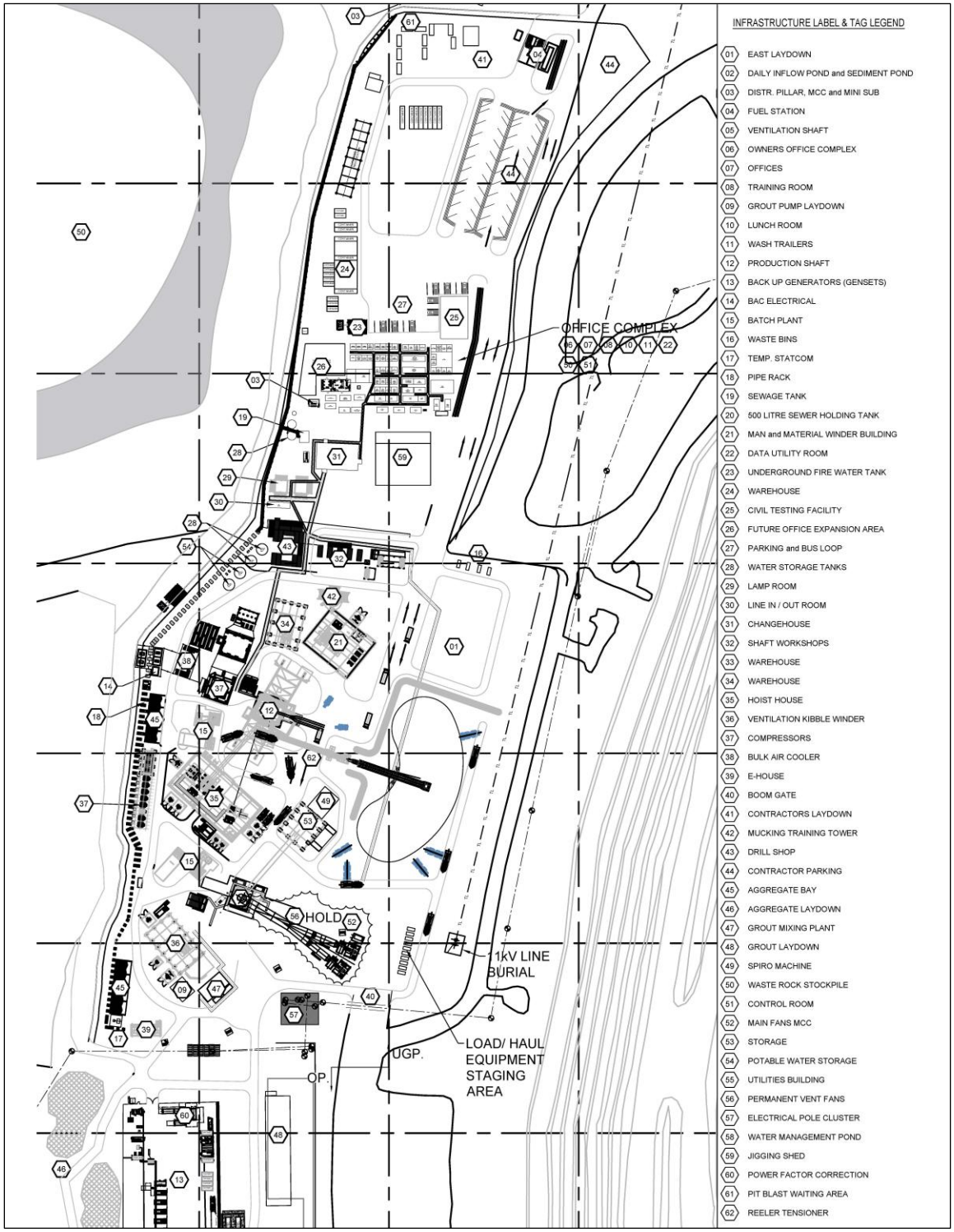
  - 132/11 kV substation and switchyard located at KDM.
- On-site Electrical:
  - Distribution of 11 kV power from the KDM substation to the Project site;
  - UGP pad surface substation, e-house and power distribution; and
  - Eight MW of diesel generator back-up power.
- Water Management:
  - Reverse-osmosis plant expansion;
  - Sewage treatment plant expansion;
  - Potable and Service water tanks and pumps;
  - Surface sediment ponds;
  - Water blending facilities to dilute high saline water before site discharge; and
  - Mechanical Evaporation Pond and partial equipment.
- Construction Facilities:
  - 200-person capacity camp;
  - UGP office complex;
  - Change house;
  - Maintenance shops;
  - Warehouses;
  - Lamp room;
  - Training and meeting rooms; and
  - Local first aid room.
- Shafts:
  - Temporary sinking headframes, hoist houses, and control cabins;
  - Permanent headframe components and hoist houses; and
  - Sinking fans, coolers, and controls.

- Fixed Plant:
  - Compressed air plant;
  - Bulk Air Cooling plant;
  - Grout plant;
  - Batch plant (2x); and
  - Firewater tanks, pumps, hydrants.

Major surface facilities remaining to be built for the UGP include the main exhaust fans, conveyors, permanent headgear rail, chutes, and bins, security gates, and sustaining pond expansions.

The general site layout of the UG pad in final configuration is shown in Figure 18-1.

Figure 18-1: UG Pad Area Infrastructure in Final State



18.2 Surface 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 report to the underground mine and must be pumped to the surface during the UG mine life. The water quality and quantity exceeds the requirements of the site water consumers and existing Reverse Osmosis (RO) treatment facilities are both undersized and unsuitable to manage this water. The water pumped from the UG mine will, in lieu of RO treatment, be disposed of by means of mechanical evaporation when water volume or quality levels exceed those that can be accepted by other consumers including neighboring mines (see Section 18.2.2). Fully lined surface ponds have been 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 and suspended solids 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 and relies heavily on the underlying water balance, chemistry, and disposal technology investigated for the 2023 Feasibility Study (Doerksen et al., 2024) with updates to hydrogeologic modelling and infrastructure as-builts.

18.2.1 Volumes and Chemical Composition of the UG Saline Water

Table 18-1 lists the estimated flow rates of saline water to be pumped to the surface by the UGP. The flow rates in Table 18-1 are taken from Itasca updated 3-D Hydrogeological Model November 11, 2025 (refer to Section 16.4). A peak groundwater inflow of 300 m^3/h is estimated to be pumped to the surface in 2031, with little change through to 2037 (see Section 16.4.7.4).

Table 18-1 also provides the salt loads and solids that will be pumped to surface from groundwater and ROM fines. Salt and solid loads are based on the estimated monthly average daily flow rates pumped to the surface each year with estimated concentrations.

Table 18-1: Estimated Volumes of UGP Water Pumped to Surface

Year	Groundwater Inflow (1)	Precipitation (2)	Service Water (3)	Total Water Pumped to Ponds		Total Dissolved Solids (4)	Total Suspended Solids (5)
	m^3 (000)	m^3 (000)	m^3 (000)	m^3 (000)	m^3/hr	m^3 (000)	m^3 (000)
2025	118	0	2	120	54	3	0
2026	482	0	9	491	56	15	2
2027	852	0	9	861	98	20	3
2028	989	0	9	998	114	22	3

Year	Groundwater Inflow (1)	Precipitation (2)	Service Water (3)	Total Water Pumped to Ponds		Total Dissolved Solids (4)	Total Suspended Solids (5)
	m^3 (000)	m^3 (000)	m^3 (000)	m^3 (000)	m^3/hr	m^3 (000)	m^3 (000)
2029	1,206	0	9	1,214	139	27	4
2030	1,713	105	9	1,827	209	41	6
2031	2,321	138	9	2,467	282	55	9
2032	2,613	138	9	2,759	315	61	10
2033	2,627	138	9	2,773	317	62	10
2034	2,635	138	9	2,781	318	62	10
2035	2,631	138	9	2,778	317	62	10
2036	2,612	138	9	2,758	315	61	10
2037	1,731	90	6	1,827	313	41	6
Total	**22,527**	**1,022**	**104**	**23,654**	**272**	**531**	**82**

Notes:

1. Inflows prepared by Itasca – refer to Section 16.4 (Itasca, 2025);
2. Precipitation via worldbank.org (Botswana), Pit catchment area per Lucara "CutDB12", runoff coefficient of 85%, crown pillar hydraulic connection to Underground Mine in 2030 as per the Mine Plan;
3. 1 m^3/hr allowance for equipment and washdown;
4. 30,000 mg/L applicable to groundwater only, 1.35 g/cm^3 crystalized density and 1.20 density in solution with maximum solution concentration of 300,000 mg/L; and
5. 1% allowance for suspended solids in dewatering system.

The chemical composition of the UG saline water has been collected at regular intervals since shaft sinking commenced in 2022. Samples of deep water were collected from two (2) deep surface boreholes (Exigo1 BH and Vent Shaft BH), as well as from grouting and probe boreholes from underground. 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 and 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 significant concentrations for other constituents of the UG water. It is noted that the UG water temperature (measured in the field at the time of sampling) was consistently at 40° 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	TDS	Ca	Mg	Na	Cl	SO4	Fe
		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
7/4/2022 Field	7.48	26,900	2848	170	7,428	17,995	286	0.17
18/03/22 VS BH	7.69	25,074	3776	34	7,000	17,995	594	1
13/07/22 VS BH	7.56	27,846	3168	131	8,000	18,370	595	1.6
28/07/22 VS BH	7.36	32,266	2968	301	3,333	11,000	622	0.95
28/07/22 Exigo1	7.29	31,764	2496	389	3,333	10,000	521	0.48
29/08/22 Exigo1	7.48	31,255	-----	-----	-----	13,696	610	-----
29/08/22 Exigo1	7.54	31,120	-----	-----	-----	14,497	544	-----
30/08/22 Exigo1	7.47	32,125	-----	-----	-----	13,996	596	-----
30/08/22 Exigo1	7.63	30,705	-----	----	-----	13,796	559	-----
20/09/22 Exigo1	7.39	28,832	----	---	----	16,146	722	----
21/09/22 Exigo1	8.25	33,048	2826	36	6,651	18,895	1006	0.64
10/04/22 Third Party Lab UGP BH	7.4	30,464	3726	32	6,234	----	----	<0.02
11/07/22 Exigo 1	6.6	28,224	----	----	----	26,893	863	----
11/07/22 UGP BH	7.1	27,972	-----	-----	----	18,895	863	-----
30/11/22 Exigo 1	7.6	24,759	3500	72.9	9,000	18,000	665	0.14
02/12/22 Exigo1	7.7	24,696	3560	58.4	8,000	17,620	417	0.08
21/05/24 NE Prod Shaft	6.9	29,990	----	----	----	18,231	373.5	0.88
07/06/24 549 Probe Holes	7	31,070	----	----	----	21,104	549.8	1.05
21/06/24 773 Level	7.1	28,220	----	----	----	14,976	963	2.51
21/06/24 V/S SubStn	7.4	32,370	----	----	----	15,592	203	2.46

Date/ Location	pH	TDS mg/L	Ca mg/L	Mg mg/L	Na mg/L	Cl mg/L	SO4 mg/L	Fe mg/L
27/06/24 V/S 548 Access	7.24	29,550	----	----	----	14,980	316	0.73
25/07/24 470L Access	6.87	30,390	----	----	9,860	17,994	442	3.04
30/07/24 773L Sump	12.25	8,714	53.6	0.06	----	30,54	572	----
15/08/24 P/S 604.5 m	7.22	28,900	----	----	6,920	16,570	706	2.86
15/08/24 Access003	6.69	31,690	----	----	10,680	18,168	543	3.32
24/08/24 x3 Cover Drill	6.83	8,867	----	----	3,500	4,380	137	0.13
24/08/24 x2 Cover Drill	6.29	32,700	----	----	13,720	18,482	410	0.88
24/08/24 x4 Cover Drill	6.97	33,350	----	----	14,650	18,157	460	1.36
28/08/24 V/S x3 Longhole Drill	6.84	31,200	----	----	11,200	18,220	533	1.24
28/08/24 V/S x4 Longhole Drill	7.28	32,380	----	----	11,608	17,916	503	3.50
26/09/24 BH 1773	6.9	33,630	----	----	11,500	17,021	930	0.82
26/09/24 773-1	6.42	32,384	----	----	11,800	17,280	1000	1.43
3/10/2024 V/S Cover Drill	6.97	33,142	----	----	10,970	18,732	500	1.19
8/10/24 773-2	7.16	6,884	----	----	1,990	3,179	580	1.98
10/10/24 V/S Cover Drill X1	6.61	35,690	----	----	11,600	19,660	440	1.67
10/10/24 V/S Cover Drill X3	6.73	35,300	----	----	10,400	19,475	450	0.92
15/10/24 P/Hole 15	7.06	31,484	1500	28	11,700	----	410	----
15/10/24 P/Hole 17	7.17	31,712	1700	30	11,000	----	430	----
23/10/24 P/Hole X16 470	5.84	33,932	3000	29	11,100	----	488	----
12/11/24 V/S OP1	7.64	36,992	2805	25.5	8,350	15,203	390	----

Date/ Location	pH	TDS mg/L	Ca mg/L	Mg mg/L	Na mg/L	Cl mg/L	SO4 mg/L	Fe mg/L
12/11/24 V/S OPZ	7.22	45,412	5230	17.3	12,700	22,290	430	----
12/11/24 V/S OP4	6.67	46,766	4380	17	8,930	22,952	480	----
12/11/24 V/S OP10	6.81	46,006	6000	19.8	12,200	23,531	410	----
12/11/24 X1 HOLE	6.56	43,636	3790	20.5	10,700	20,275	500	----
29/11/24 821 Level	7.33	28,014	3190	56	8,106	12,558	420	----
29/11/24 BH797 DRILL 821 LEVEL	6	28,582	2950	80	8,093	12,647	470	----
15/01/25 PSGTDD001	7.24	32,202	2070	20.3	10,100	17,239	430	0.30
03/03/2025 CNV1	6.4	33,770	3940	34	8,280	19,140	410	1.70
27/03/25 VS Access 2	5.45	39,078	5320	29	11,300	23,000	530	0.05
27/03/25 PS Station 4	5.2	40,678	5940	31	12,100	23,000	530	0.05

Source: Betach (pty) Ltd. (2022-2025)

18.2.2 Overview of the Site Water Circuit

The KDM process plant water circuit in Figure 18-2 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 infrastructure 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 depressurize the OP for operations purpose. The dewatering of the OP brings to the process plant an average of 216 m^3/h of water (24-hour averaged hourly pumping rates). This water from OP dewatering is fed to the process plant (refer to Figure 18-2) together with water recovered from the Fine Residue Deposit (FRD) to sustain its water balance by compensating for the following losses of water:

- Water entrained by the deposited slimes in the FRD and including seepage losses from the facility;
- Water captured by the coarse tailings;

- Domestic use of water and potable water; and

- Water lost by evaporation from the different water storage infrastructure at the process plant.

Under the constant supply of the OP dewatering water, 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 90 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. This twelve-month agreement is updated annually and allows for up to 150 m^3/hr to be delivered to the Orapa Mine.

Salt Balance Within the Process Plant Water Circuit.

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 FRD (average 103 m^3/h refer to Figure 18-2), 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;

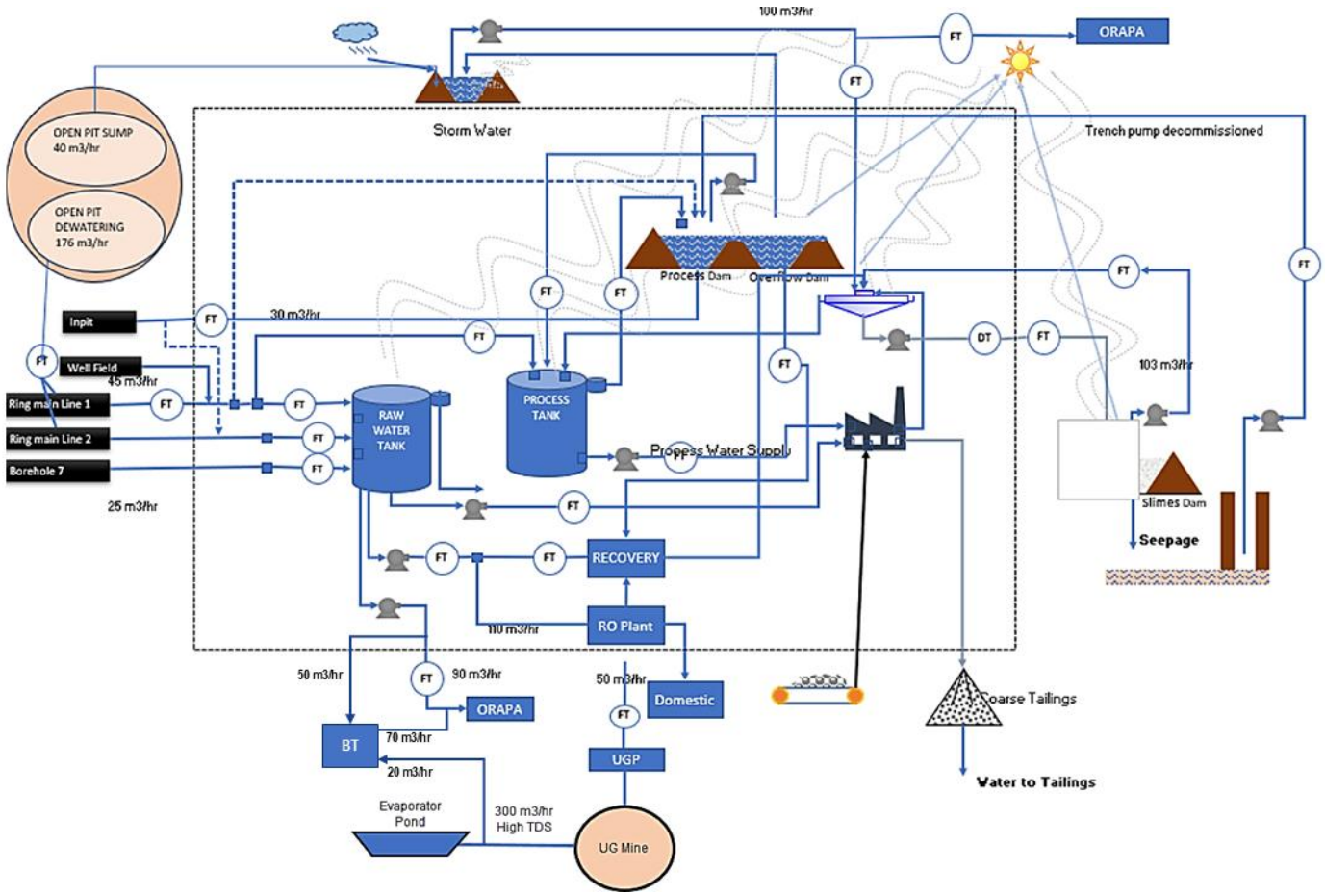
2. TDS retained with the pore water in the FRD; and

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.

A key principle in the management of the volumes of UG saline water is that none of this water can or 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 18-2 showing that the management of the UG saline water will be done primarily outside of the process plant water circuit by use of evaporation ponds, with limited discharge of high saline groundwater after blending with lower saline water from the process water tank.

Figure 18-2: Site Water Circuit



Source: Lucara (2025)

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 wells, sumps, and stormwater dams. The largest contributor is the dewatering well field around the open pit. Under these conditions the mine currently operates at a surplus and discharges upwards of 90 m^3/hr to Orapa Mine.

Per Section 16.4.4 the dewatering wells are expected to decrease to approximately 60 m^3/hr by mine closure, a reduction of 116 m^3/hr from current rates. This may result in a water shortfall at the process plant, necessitating treatment of up to 26 m^3/hr of underground mine discharge, within the plant's capacity. However, during the rainy season (November–March), stormwater collected by the 75 ha FRD is anticipated to collect significant volumes of stormwater and support a positive 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 be managed throughout the mine life. Surface discharge at the mine site of such considerable volumes (up to 315 m^3/h) of water at some 30,000 mg/L TDS is unacceptable within best water management practices, as per Botswana permit regulations, flooding of mine site areas, infiltration and salinization of groundwater upper aquifer used by farmers.

Trade-off studies performed in 2022/2023 evaluated the use of four (4) different disposal methods including local deposition to salt pans, reverse osmosis, solar evaporation, and mechanical evaporation (Doerksen et al., 2024). Mechanical evaporation was selected as the most suitable, flexible and adaptive option.

Mechanical evaporation at mine sites is a proven technology under dry arid climatic conditions as those at KDM and is used in several other applications regionally and throughout the world to eliminate excess water from the FRD, OP dewatering, and brine from RO treatment. Figure 18-3 illustrates a conceptual mechanical evaporator general arrangement.

Figure 18-3: Example Arrangement of Mechanical Evaporator



Source: African Mining Brief (2014)

The application of Mechanical Evaporation requires the construction of a pond where the mechanical evaporators are installed and whose functions are to:

1. Receive the saline water pumped from the UG mine;

2. Feed the mechanical evaporators; and

3. Provide storage for the salts.

Evaporator ponds will be constructed in a phased approach to delay capital expenditure and facilitate ongoing engineering adjustments. It is expected that pond design criteria may change as underground mine development progresses, groundwater inflow rates are monitored, and hydrogeological models are refined. Figure 18-4 provides a depiction of the planned pond phasing. The initial pond was permitted in 2024 and constructed in 2025 (refer to Section 20 for

permitting details). This pond has a total capacity of 327,450 m^3, inclusive of freeboard, and is designed to accommodate up to 191,013 m^3 of solids and 81,863 m^3 of suspended solution.

The size of the pond is governed by the following criteria:

1. Evaporators to be placed around the perimeter spaced no less than 30 m apart;

2. 1.5 m minimum depth of water required for the operation of pumps to feed the evaporators, and 50 m clearance for placement of the pump barge

3. 1.0 m minimum freeboard

4. Quantity of salts in solution and solid state to be deposited; and

5. Quantity of solids to be deposited

Each pond is capable of supporting up to 15 mechanical evaporators. A peak requirement of 14 mechanical evaporators is predicted based on vendor provided efficiencies and flow rates, and working a maximum average 12.2 hours per day from dawn to dusk. Table 18-3 outlines the predicted evaporator operating schedules.

Table 18-3: Evaporator Pond Schedule

Item	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Inflows															
Groundwater	m^3 (000)	23,802	126	580	1,035	974	1,430	1,767	2,759	2,749	2,687	2,679	2,655	2,622	1,739
Precipitation	m^3 (000)	1,022	-	-	-	-	-	105	138	138	138	138	138	138	90
Service Water	m^3 (000)	104	2	9	9	9	9	9	9	9	9	9	9	9	6
Total	m^3 (000)	24,928	128	588	1,044	983	1,439	1,881	2,905	2,895	2,834	2,826	2,801	2,769	1,835
	m^3/hr	274	15	67	119	112	164	215	332	330	323	323	320	316	209
Total Dissolved Solids															
Concentration	mg/L (000)	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Total Added	m^3 (000)	559	3	18	23	22	32	42	65	64	63	63	62	62	41
Phase 1 Pond	m^3 (000)	167	3	18	23	22	32	42	26	-	-	-	-	-	-
Phase 2 Pond	m^3 (000)	201	-	-	-	-	-	-	38	64	63	36	-	-	-
Phase 3 Pond	m^3 (000)	192	0	0	0	0	0	0	0	0	0	27	62	62	41
Total Suspended Solids															
Concentration	%	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total Added	m^3 (000)	86	-	2	4	3	5	6	10	10	10	10	10	10	6
Phase 1 Pond	m^3 (000)	24	-	2	4	3	5	6	3	-	-	-	-	-	-
Phase 2 Pond	m^3 (000)	31	-	-	-	-	-	-	7	10	10	5	-	-	-
Phase 3 Pond	m^3 (000)	30	-	-	-	-	-	-	-	-	-	5	10	10	6
Mechanical Evaporators															
Evaporation efficiency	%	55	68	55	55	55	55	55	55	55	55	55	55	55	55
Evaporator Flow Rate	m^3/hr	144	144	144	144	144	144	144	144	144	144	144	144	144	144
Evaporators Required	#	14	2	3	6	5	8	9	14	14	14	14	14	14	14
Hours per Evaporator	hr/day	8	7	8	8	6	7	8	9	8	7	7	7	7	8

Figure 18-4 shows the siting of the existing and future ponds, located at some 1 km from the shaft area. 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.

Saline water will be pumped from the underground mine and deposited into the North side of the pond, which is graded to the south, with a pump barge located on the South end of the pond to feed the evaporators. This arrangement will encourage solids to drop from the solution before reaching the pump barge. Mechanical evaporators will be situated along the East, West, and South side of the pond and spray back into the pond.

The initial TDS concentration of pond water is anticipated to be 30,000 mg/L. As water is evaporated and salt is retained a progressive increase in TDS within the pond will occur. It is expected that the TDS concentration in the pond will reach a saturated level of approximately 400,000 mg/L within one year of operation (Pineau, 2023) and will maintain itself at this level for the subsequent years of Operation. Once the saturated concentration is reached, salts will start crystallizing out of the solution and deposit at the bottom of the Pond.

As the pond fills additional ponds will be constructed with mechanical evaporators relocated to allow continued deposition. Closure to the pond will be part of the overall closure plan of KDM.

Figure 18-4: Evaporator Pond Phasing

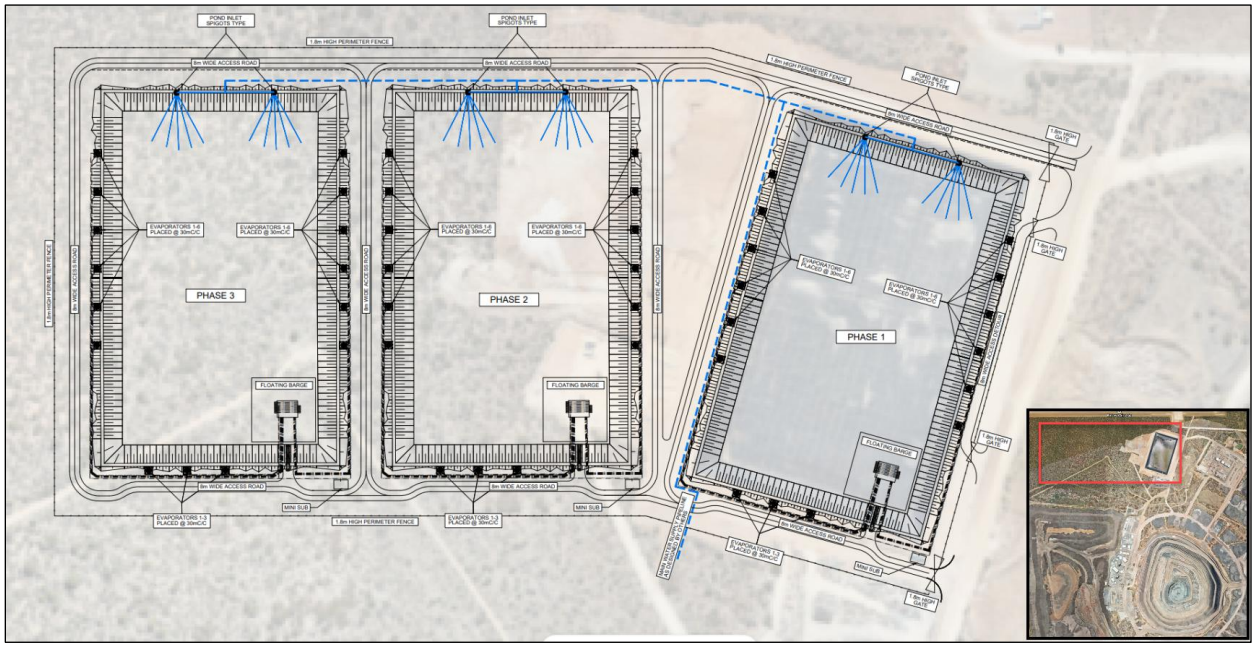
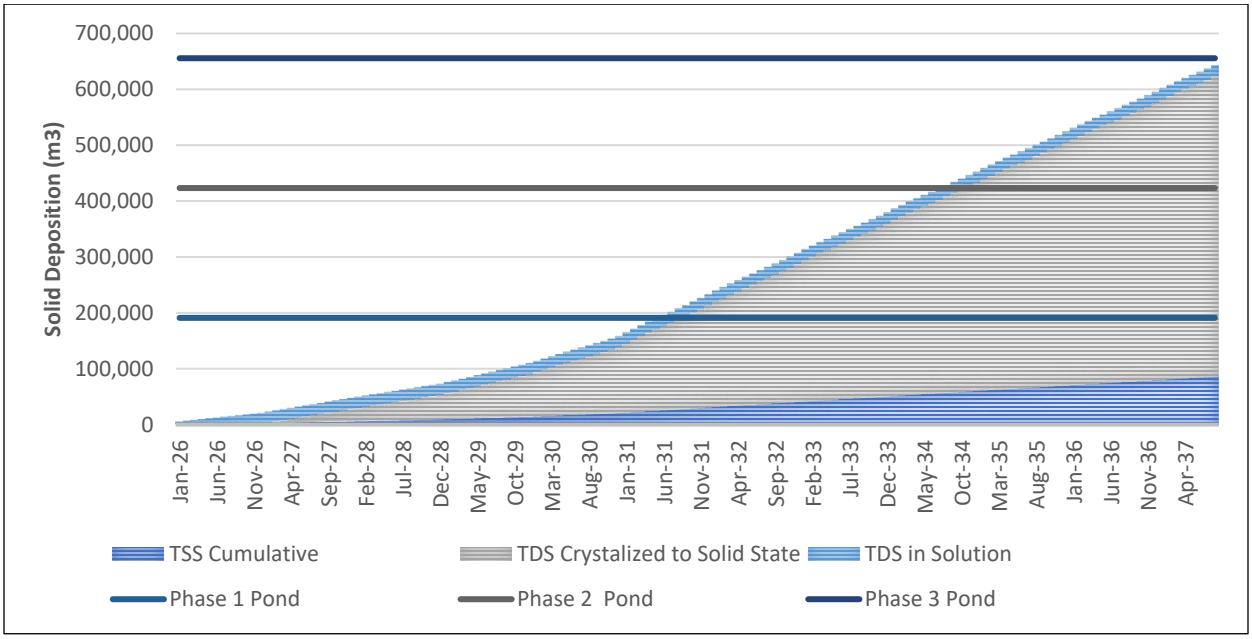


Figure 18-5: Evaporator Pond Solids Deposition and Phasing



18.3 Waste Rock Management

Waste rock generated from both OP and UG development activities is hauled to a designated surface waste rock storage facility (WRSF) located on the west side of the mine property. The facility has sufficient capacity to accommodate all remaining OP and UG waste as per the mine plan. Survey monitoring points have been installed at strategic locations across the dump to track any potential movement or settlement; to date, no significant movement or settlement has been observed. All host waste rock at the KDM site has been classified as non-acid generating and is not expected to contribute to acid rock drainage or metal leaching. Consequently, no special treatment is required for the waste rock, with management focused on ensuring geotechnical stability in accordance with engineered design specifications.

18.4 Residue (Tailings) Storage Facilities

18.4.1 Introduction

The purpose of the Karowe Fines FRD and Coarse Residue Disposal (CRD) projects is to provide Karowe Mine with continuity in processing mineral residue for the LOM (2038). This will be achieved by providing residue facilities which will be able to contain residue material generated during the processing of the feed material. The current expansion plans for the residue facilities considered the surface area required for the total volume of material to be deposited by the end

of LOM. Presently and into the foreseeable LOM future, the facilities will have to contain 37.3 Mt of residue material (including material from stockpiles).

The Karowe Fines Residue Disposal (FRD) project consists of three (3) facilities namely, FRD 1, FRD 2, FRD 3 (Not yet constructed). A feasibility study for the design of FRD 1 lift was carried out in 2019 by Knight Piésold (KP). The study proposed increasing the height of FRD 1 to a final elevation of 1042 masl and for 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).

Following the adoption of the Global Industry Standard on Tailings management (GISTM) as best practice for residue disposal facilities by Lucara Botswana, KP was appointed to re-design the FRD 2 in 2021 taking into account GISTM and height restrictions imposed (Lucara limitation) on FRD 1 to a maximum elevation of 1031 masl. The 2021 re-design further optimised the FRD 2 site to fall within the mine fence area with a design elevation of 1026 masl. The FRD 2, which began construction of starter walls in 2022 is approximately 1100 m wide and 550 m long, with a divider wall creating two paddocks labelled as 2A and 2B (Knight Piésold, 2022). The current FRD 2 facility design can accommodate the fine residue material (tailings) up to the end of 2027.

With the introduction of the underground expansion project, the feasibility of the Phase 3 facility was further assessed in 2024. One of the key objectives of this study was to maximise the capability of FRD 2 (Phase 2) to optimise the footprint and cost of FRD 3 (Phase 3). An additional lift was therefore incorporated into the FRD 2 design to raise its crest to the same elevation as FRD 1 (1031 masl), enabling FRD 2 to store additional residue from the underground operations beyond 2027. FRD 3 was also designed to a final elevation of 1031 masl and is intended to receive residue material once FRD 2 reaches capacity, providing storage through to LOM. All three facilities designs remain compliant with the mine's 1031 masl maximum height restriction.

Contained within this section are the 2025 feasibility designs for the mine residue facilities referred to as the CRD, FRD 2 and FRD 3.

18.4.2 Design Criteria

The design criteria presented in Table 18-4 defines the project design requirements for the extension of the CRD, wall raising for the FRD 2, and the development of FRD 3. These criteria have been established in accordance with the specifications provided by Karowe Diamond Mine (KDM) and in compliance with GISTM requirements.

Table 18-4: Design Criteria for CRD Extension and the Development of the FRDs

Criteria	Units	Design	Source
Life of Facility Required	Yrs.	14 (2025 to 2038)	Lucara 2025 Projection
Run of Mine (ROM)	Mt	37.3	Lucara 2025 Projection
Tonnages of CRD and FRD Generated	Mass Percentage Split	40% to FRD and 60% of Tonnes to CRD	Karowe Diamond Mine

Criteria	Units	Design	Source
Coarse Residue Deposit (2025 – 2040)			
Tonnes to Facility	Mt	22.4	Lucara 2025 Projection
In Situ Density	t/m³	1.6	(Geoflux, 2024)
Volume for LOM	Mm³	14.0	Lucara 2025 Projection
Maximum Height	m	38	(Lucara, 2025)
Design Slope	Ratio (V:H)	1:1.5	(Lucara, 2025)
Fine Residue Deposit (All Facilities 2025 – 2040)			
Tonnes to Facility	Mt	14.9	Lucara 2025 Projection
In Situ Density	t/m³	1.35	Karowe Diamond Mine
Volume for LOM	Mm³	11.1	Lucara Projection
Maximum Crest Elevation	mL	1031	(Knight Piésold, 2024)
Impoundment Wall Inside Slope	Ratio (V:H)	1:2	(Knight Piésold, 2024)
Impoundment Wall Outside Slope	Ratio (V:H)	1:3	(Knight Piésold, 2024)
Crest Width FRD 2	m	8	(Knight Piésold, 2025)
Crest Width FRD 3	m	10	(Knight Piésold, 2024)
Area Required	Ha	156	(Knight Piésold, 2024)
Minimum freeboard	m	1	(Knight Piésold, 2024)
Consequence Classification (CCS) FRD 2		High	(Geoflux, 2025)
Consequence Classification (CCS) FRD 3		High	(Knight Piésold, 2024)

Source: KP (2025)

In order to model the footprint of the CRD at LOM, the July 2025 survey surface provided to KP was used as the base for modelling the required volume calculated with the planned production data for July 2025 to 2038. Production tonnages adopted for this study are provided in Table 18- and Table 18-6, and reflect the mining and processing plan for this study.

Table 18-5: Mill Feed from Stockpiles

Year	Annual Tonnages (Mt)		
	CRD	FRD	Total
2025	0.1	0.1	0.2
2026	1.3	0.9	2.2
2027	1.4	0.9	2.3

Year	Annual Tonnages (Mt)		
	CRD	FRD	Total
2028	0.2	0.2	0.4
2029	0.0	0.0	0.1
2030	0.0	0.0	0.1
2031	-	-	-
2032	-	-	-
2033	-	-	-
2034	-	-	-
2035	-	-	-
2036	-	-	-
2037	-	-	-
2038	1.4	0.9	2.4
2039	-	-	-
2040	-	-	-
		Total	**7.7**

Source: Lucara (2025)

Table 18-6: Direct Mill Feed

Year	Annual Tonnages (Mt)		
	CRD	FRD	Total
2025	0.3	0.6	0.5
2026	0.4	0.2	0.6
2027	0.3	0.2	0.5
2028	1.5	1.0	2.4
2029	1.7	1.1	2.8
2030	1.7	1.1	2.8
2031	1.7	1.1	2.9
2032	1.7	1.1	2.9
2033	1.7	1.1	2.9
2034	1.7	1.1	2.9
2035	1.7	1.1	2.9
2036	1.7	1.1	2.9
2037	1.7	1.1	2.9
2038	-	-	-

Year	Annual Tonnages (Mt)		
	CRD	FRD	Total
2039	-	-	-
2040	-	-	-
		Total	**29.6**

Source: Lucara (2025)

18.4.3 Geotechnical Investigations

18.4.3.1 Site Specific Investigations

18.4.3.1.1 FRD 2

Geotechnical investigations were carried out to determine the properties and the extent of the soils and whether there is enough suitable material to construct the first lift of FRD 2. 28 test pits were excavated, seven of them being within the FRD footprint. The profile at the site generally consists of orangey brown slightly silty sand with an average thickness of 1.8 m overlying calcrete in the form of powder and nodules. No groundwater was observed in any of the test pits, the walls were stable, and the maximum depth for the test pits was 3.8 m. Refusal occurred in some test pits at depths between 1.6 m and 2.3 m below the surface. Samples of the materials from the site area were sent for laboratory testing. These tests included: California Bearing Ratio, Modified AASHTO Proctor density and moisture content tests, Specific Gravity and Dispersity (Knight Piésold, 2022).

18.4.3.1.2 FRD 3

Investigations included geophysical surveys (ERT and MASW), 33 test pits, 9 rotary core boreholes, 7 groundwater monitoring boreholes (percussion drilling and test pumping), and laboratory testing on representative samples. The surveys revealed a heterogeneous subsurface comprising loose aeolian sands, nodular and hardpan calcrete, and dense silcrete. Depth to competent material varies significantly, with localised fractures, weathering zones, and dolerite intrusions.

Soils: Aeolian silty sands (G7) and calcareous silty sands (G8) dominate the upper profile, underlain by nodular calcrete (G7). These materials are of low to very low permeability (10^{-6} to 10^{-9} m/s), non-dispersive, and collapse susceptible. Hardpan silcrete shows strong to very strong UCS values (up to 110 MPa). Triaxial tests confirmed non-brittle, strain-hardening behaviour for aeolian sands.

FRD 3 soils are collapsible therefore loose sands must be compacted before starter wall construction. Suitable construction materials (G7/G8 sands and calcrete) are available on site, with estimated surpluses relative to requirements. Basalt aggregates provide durable rockfill. Excavation conditions are mostly “soft” to ~4–5 m depth, transitioning to “intermediate” and “hard”

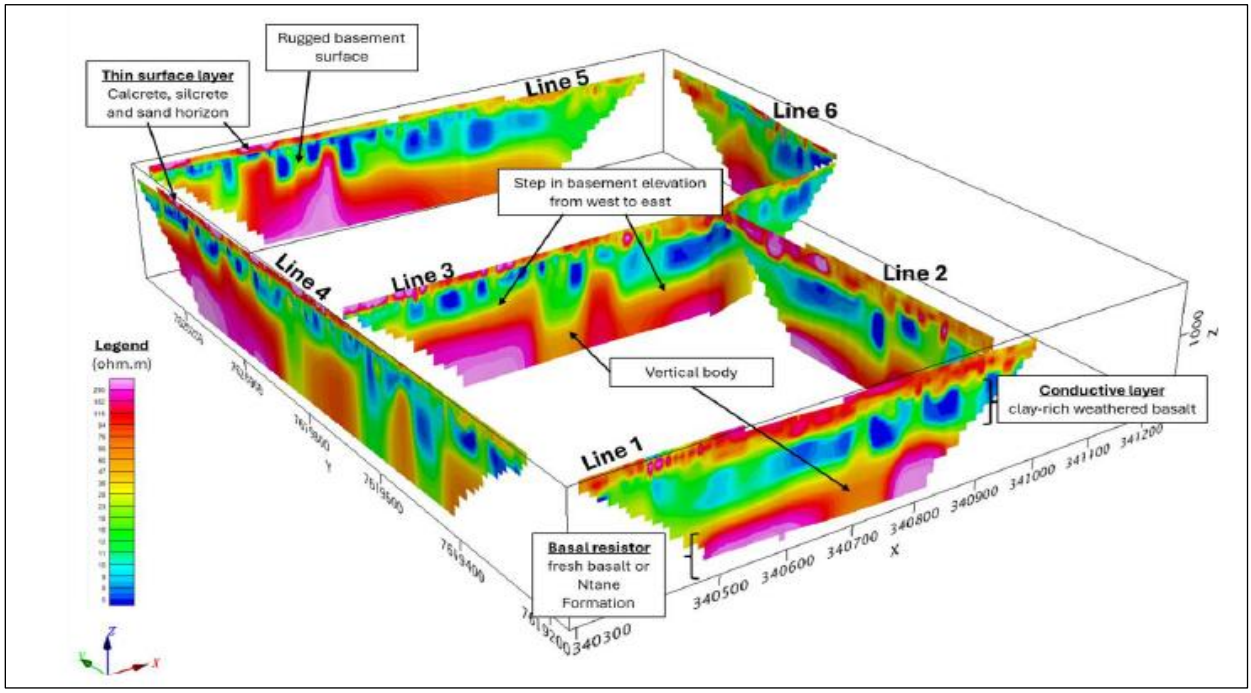
at depth or near site margins. Hydraulic hammers/rippers may be needed locally (Knight Piésold, 2024).

Hydrochemistry: Large gaps in the water quality data were noted. The groundwater quality data is dominated by sodium (Na) and potassium (K) cations and by chloride (Cl) anions. Samples collected from the pit and FRD demonstrate elevated levels of major ions, showing seepage from the FRD. The wellfield boreholes (Ntane Sandstone) show a very different, signature from that of the FRD and deeper groundwater boreholes. The Na-K and Cl ions dominate the deeper groundwater water samples and are regarded as sodium chloride water type, typical of saline water.

Hydrogeology: The main water-bearing zones are the Stormberg Basalt-Ntane Sandstone contact, Fracture Corridor, Mosolotsane base, and Northern Kimberlite Pipe. Dolerite dykes act as groundwater barriers, but the aquifers are not entirely compartmentalised. A perched water table can develop in the calcrete/silcrete unit, particularly in enhanced recharge zones.

Geophysics: The survey data suggests the following near-surface stratigraphy: The topmost layer consists of calcrete, silcrete, and sand, which is unevenly cemented. The middle layer is weathered and clay-rich basalt, and the bottom layer is fresh basalt. Vertical features such as faults, fracture zones, and dykes disrupt these layers. Geophysics sections were incorporated into 3D interpretation using Leapfrog Works®. The identified targets were used for the proposed drilling and pump testing program.

Figure 18-6: 3D model of the ERT survey results

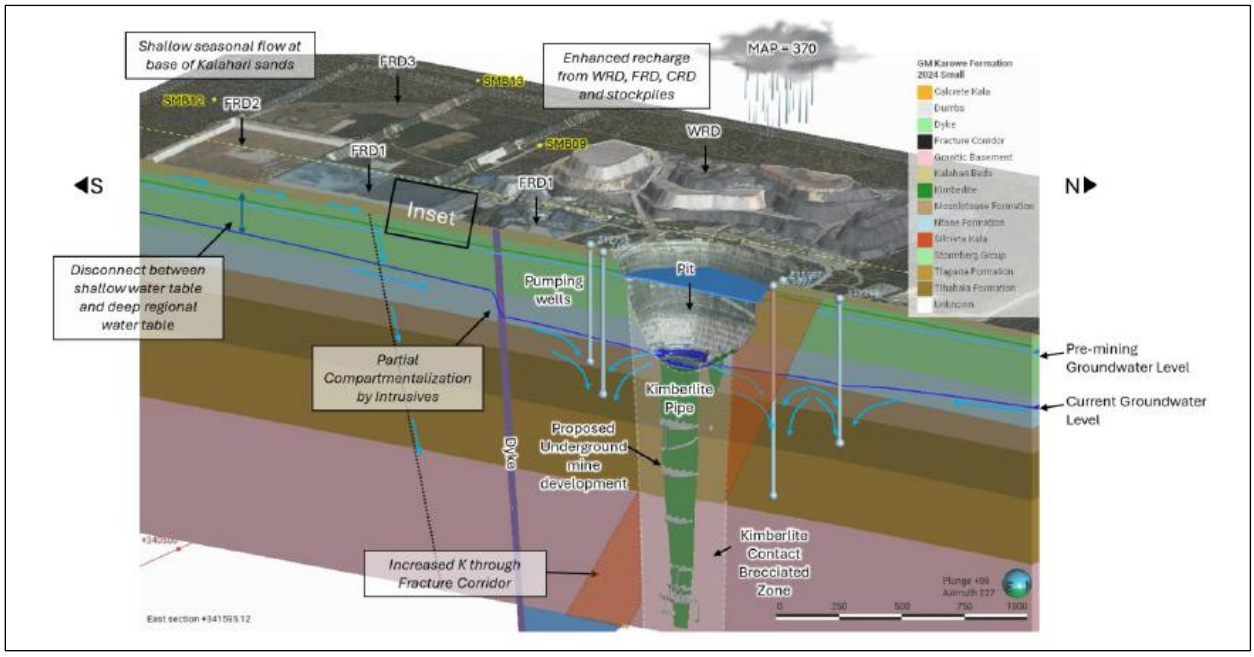


Source: KP (2024)

Drilling and Aquifer Testing: One borehole was drilled north of the FRD 3 site for monitoring and constructed to prevent seepage from the FRDs. Due to the risks of drilling into the regional aquifer, KP was asked to replace deeper holes with shallow monitoring ones around the CRD. Aquifer testing was done on boreholes with water strikes >1 m^3/hr, and pumping tests were conducted on SMB09, CRD001, and CRD003 from Dec 9, 2023, to Feb 16, 2024.

Hydrogeological Conceptual Model: Seepage from the FRDs may migrate laterally in a northerly direction and then drawn into the pit due to the hydraulic gradient due to the cone of drawdown. Flow into the underlying unsaturated basalts is slow as most of it goes into storage. The Fracture Corridor, Kimberlite contact fractured zone, and dyke contacts enhance groundwater flow through the Karoo sediments. However, the dykes appear to be partially compartmentalising the regional aquifer and inhibiting groundwater movement towards the pit. Dewatering of the underground workings will create a steep hydraulic gradient, driving regional groundwater towards the pit and the kimberlite pipe. The Mine's dewatering and reticulation circuit will capture seepage that seeps through structures in the overlying basalt.

Figure 18-7: Conceptual Model of Karowe Mine

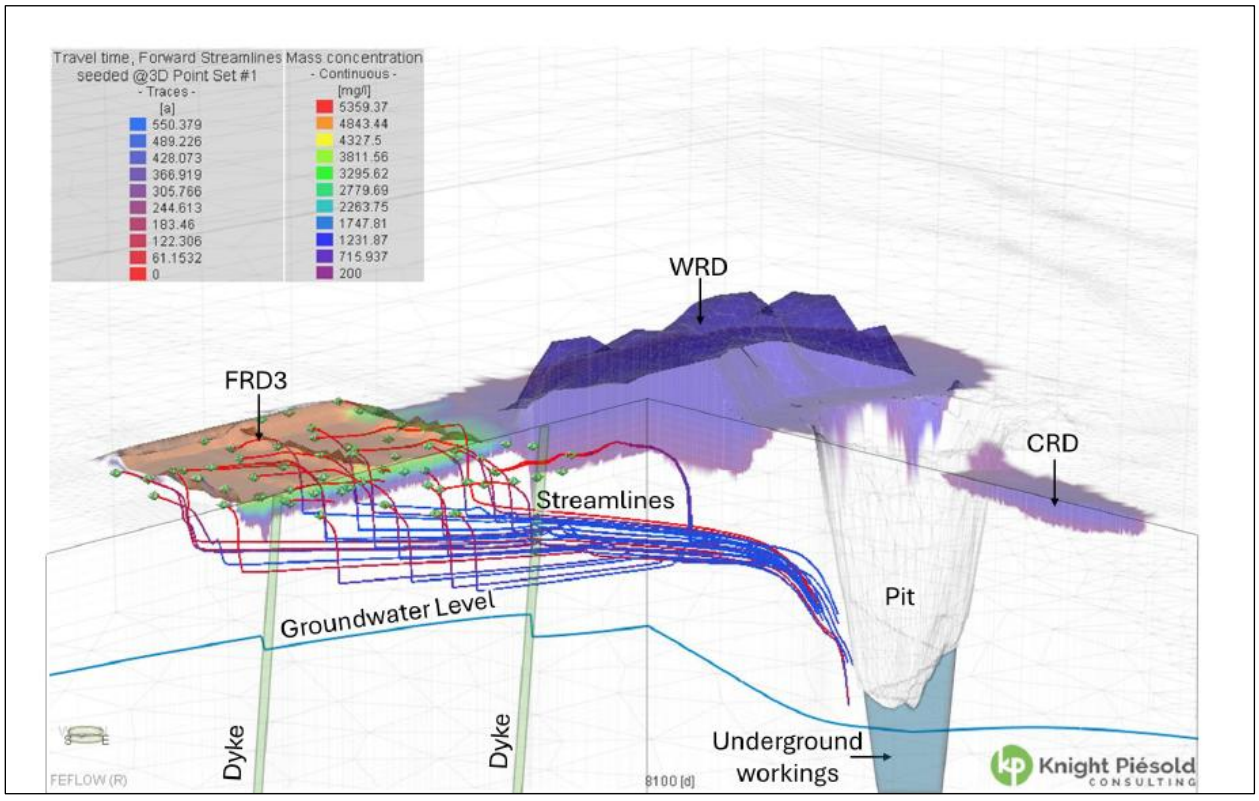


Source: KP (2024)

Hydrogeological Numerical Model: Pit dewatering was simulated from the initial step, and underground dewatering wells were modelled to activate only when the underground mining commences. Groundwater streamlines are trajectories of particles following the velocity vectors of the flow based on a steady-state flow field to predict the time and direction that fluid could travel from a point source.

Post-operation conditions where all active dewatering ceases and the underground voids and open pits are allowed to flood, will result in rebounding of the groundwater level, likely stabilising lower than the pre-mining water table due to evaporation. This will result in the formation of a pit lake, as the open pit continues to act as an evaporative sink. The model simulates that the groundwater flow with seepage from the FRD 3 and other FRD extensions will continue to migrate towards the pit but at a slower rate than during the mining operations, due to the lower hydraulic gradient as the pit lake level recovers.

Figure 18-8: Streamlines and Chloride contamination plume



Source: KP (2024)

Seismicity: Peak ground accelerations (PGA) and spectral accelerations provided by the PSHA are provided for a time-averaged shear wave velocity (Vs30) of 760 m/sec in the top 30 m of the foundation profile (representing rock site conditions). Site-specific ground motion parameters are also provided which account for the foundation conditions underlying the proposed FRD 3. A Vs30 value of 520 m/sec was used, representing the lower range of values identified at some locations, including the southwest corner of FRD 3. The calculated PGA values for a Vs30 of 520 m/sec are approximately 10% higher compared to those calculated for a Vs30 of 760 m/sec. The PGA values for a return period of 10,000 years are approximately 0.16 g and 0.17 g for Vs30 values of 760 m/sec and 520 m/sec respectively.

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 utilises a single conveyor system, which builds the CRD from North to South and a bypass conveyor that builds from East to West which also acts as the standby when the conveyor system is being extended. In the 2019 feasibility study, the CRD deposition recommended a three-leg conveyor system to optimise the storage to space ratio as well as adding some redundancy. Leg 1 conveyor, according to the December 2022 survey, was terminated at approximately 1200 m from the start of the conveyor system.

As of July 2025, Leg 2 of the conveyor system will continue depositing on its current leg in a southerly direction until it reaches its final position. After it has completed the first leg, it will be relocated to the third and deposit there. 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.

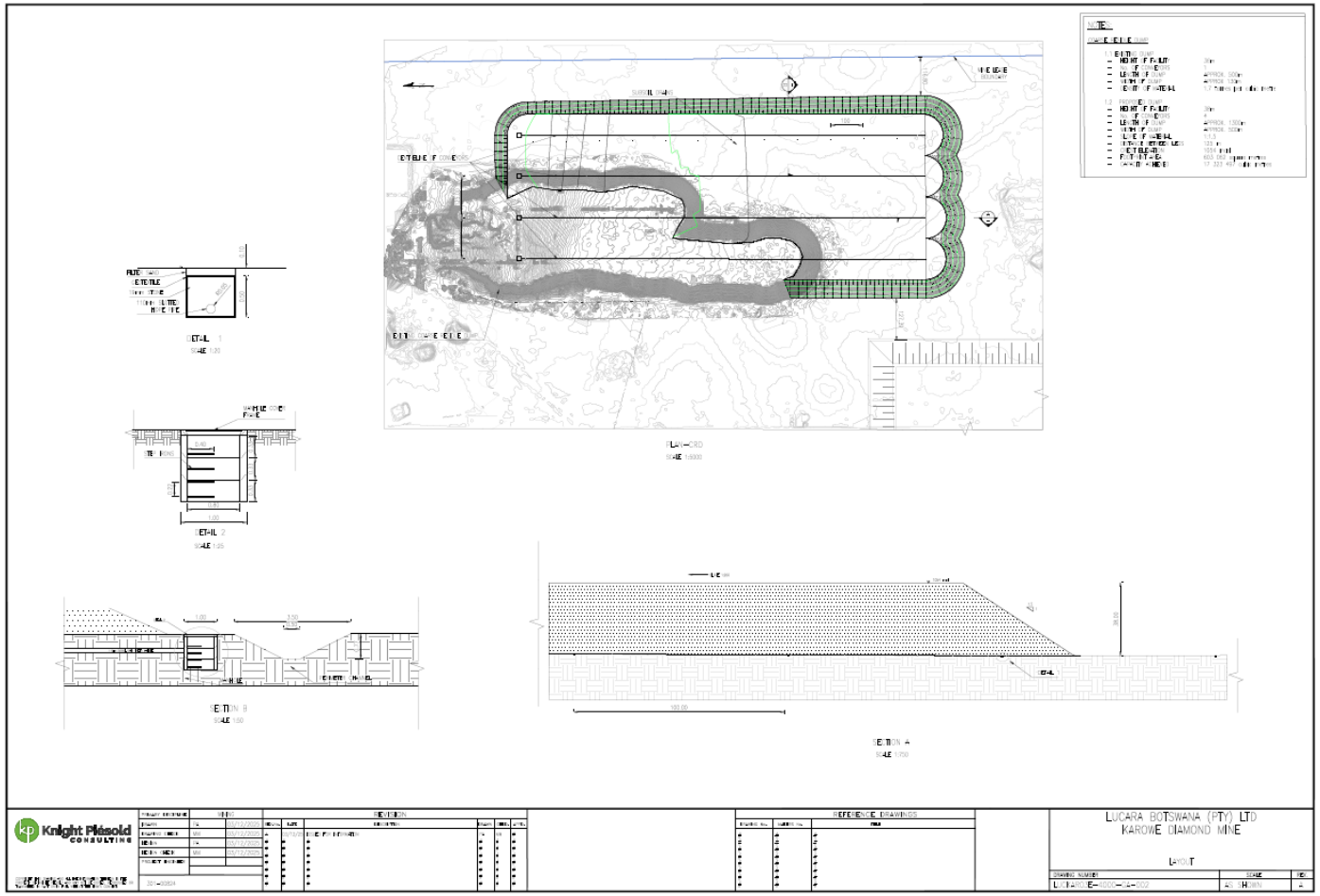
-The existing conveyor system would need to continue depositing on its current leg in a southerly direction until it reaches the final revised position as indicated Figure 18-9, which is roughly 1300 m from the start of the conveyor system. To achieve the current LOM tonnage requirements, an additional fourth leg is necessary. This need arises primarily from two factors:

Change in deposition strategy: The 2019 capacity analysis focused on having a higher ramp elevation thus elevating the dump. However, the 2025 plan suggests adding a fourth leg to increase capacity and reduce the rework needed to elevate the dump; and

- Increased production output: Between 2019 and 2025, production output has consistently increased. The originally required LOM capacity for the CRD was 14.3 Mm³ in 2019 and 13.1 Mm³ in 2024. Despite ongoing deposition, the capacity requirement has increased, reaching 14.0 Mm³ as of 2025 projections.

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 production forecast, 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 18-9: CRD Model



Source: KP (2025)

In addition to accommodating tonnages to 2038, 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 monitored as the CRD development progresses. The CRD model results have been summarised in Table 18-7.

Table 18-7: Summary of Proposed CRD Facility Design Characteristics

Parameter	Units	Value
Design Life (2025 – 2038)	Years	14
Total Storage Required (2025 – 2038)	Mm³	14.0
Total Storage Achieved	Mm³	17.3
Crest Elevation	m	1054
Maximum Height of Facility	mL	38
Number of Conveyor Legs	No.	4
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 Fence	m	Approx. 115
Distance from Landfill Sites from CRD toe on the southern extremity	m	Approx. 140

Source: KP (2025)

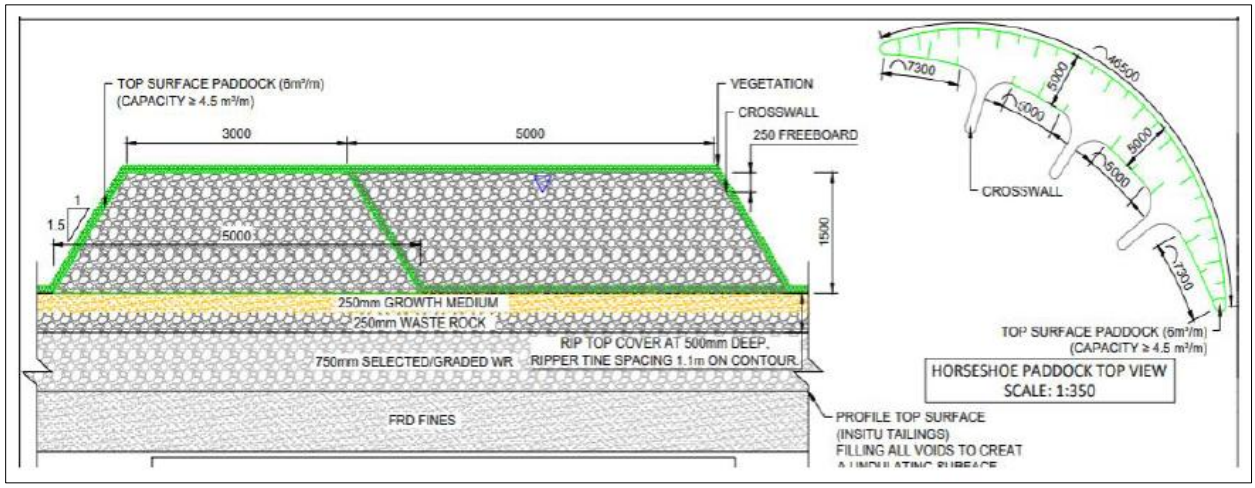
18.4.5 Fine Residue Deposit (FRD)

Contained within this section are the feasibility designs for the slimes dams called the Fine Residue Deposits (FRD) facilities. At the time of this Report, there are only two existing facilities (FRD 1 and FRD 2), with the third one (FRD 3) planned to be constructed for future operations. Currently, FRD 1 is under care and maintenance with closure plans being finalized by E-Tek Consulting. Deposition is on-going on FRD 2, with the wall lift construction activities and deposition being rotated between the two paddocks.

18.4.5.1 FRD 1

The following extracts are summarised from the E-Tek Consulting (Pty) Ltd closure report (E-Tek Consulting (Pty) Ltd, 2024). The closure strategy for the FRD 1 aims to ensure long-term stability, erosion control, and sustainable post-mining land use. The facility will be reshaped using imported waste rock to achieve a gentler final slope (18°) without compromising wall integrity. As per Figure 18-10 a growth medium of calcrete and topsoil will be applied, supported by a layered cover system to manage salinity, limit capillary rise, and promote vegetation establishment (E-Tek Consulting (Pty) Ltd, 2024)

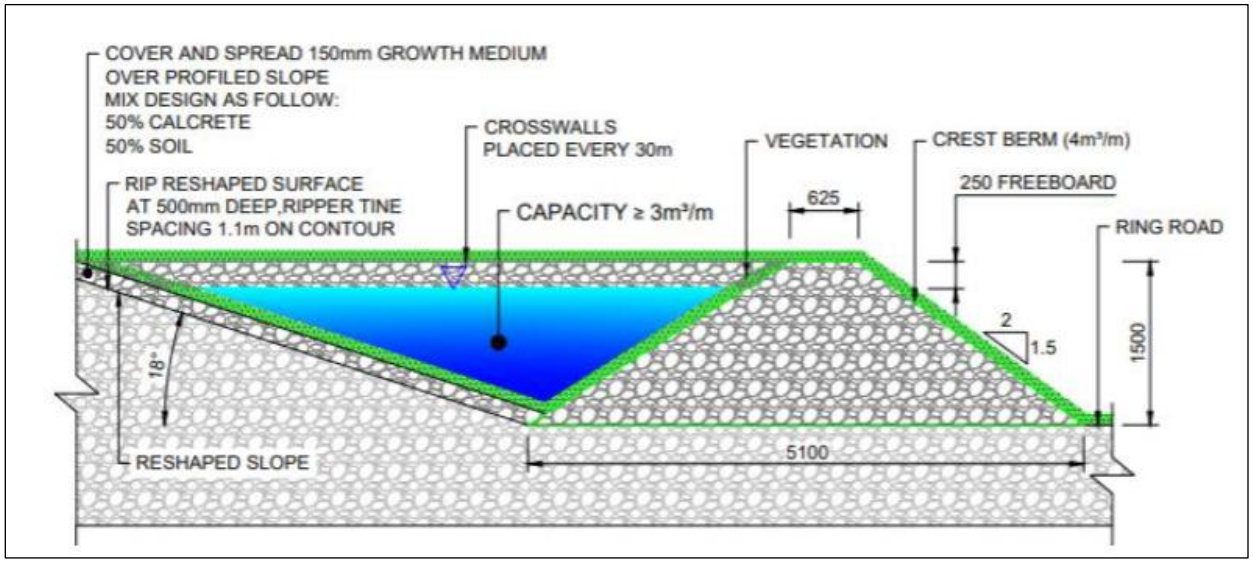
Figure 18-10: Proposed Design of the New Horse-Shoe Paddock



Source: E-Tek (2024)

Stormwater management is integrated into the design to minimise erosion and retain water for vegetation in the semi-arid climate as shown on Figure 18-11. Indigenous grasses, shrubs, and trees will be planted to create a stable and biodiverse landform. (E-Tek Consulting (Pty) Ltd, 2024).

Figure 18-11: Section of Toe Berm



Source: E-Tek (2024)

Post-closure monitoring will cover surface and groundwater quality, vegetation success, air quality (dust), and stability to ensure that closure objectives are sustained. Risks such as slope instability, groundwater contamination, and drought impacts were assessed, with mitigation measures included in the rehabilitation plan (E-Tek Consulting (Pty) Ltd, 2024).

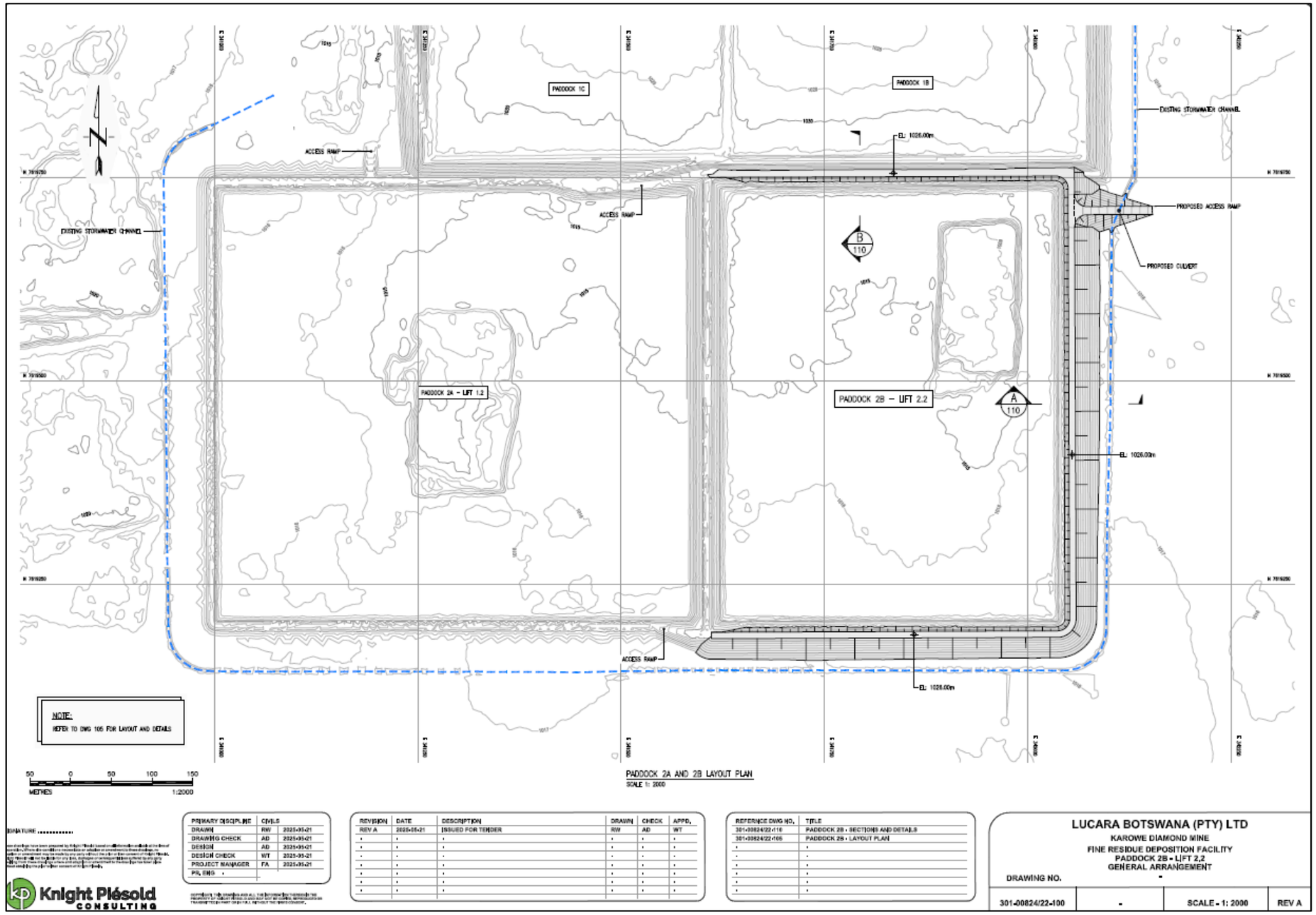
According to E-Tek, the overall closure design aligns with GISTM requirements, ensuring compliance with safety, environmental, and social commitments while supporting a sustainable post-mining landscape (E-Tek Consulting (Pty) Ltd, 2024).

18.4.5.2 FRD 2

The current deposition method 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 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 the 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 undergoing closure, the current design for the FRD 2 facility allows for deposition and wall raising. The FRD 2 facility has been designed to accommodate deposition up to 1026 masl, with potential expansion to 1031 masl if required. The figure below presents the current general arrangement (GA) for FRD 2., the GA shows the latest lift design to raise Paddock 2B embankment wall to 1026 masl.

Figure 18-12: FRD 2 General Arrangement



Source: KP (2025)

18.4.5.3 FRD 3

The FRD 3 facility has been designed to be raised progressively to a final elevation of 1031 masl, with the starter wall constructed from compacted layers of selected calcrete in accordance with site-specific standards. To improve pool control and water management, the facility will be divided into two paddocks. The impoundment walls incorporate a 10 m wide crest, providing safe access and traffic management along the perimeter of the facility. For long-term stability, the downstream and upstream slopes are designed at gradients of 1V:3H and 1V:2H, respectively.

The FRDs accumulated process water, together with any stormwater, will be removed through a decant system comprising of pump units connected to the turret system and conveyed to the stormwater dam. The decant water from the dam will be pumped back directly from the facility to the plant.

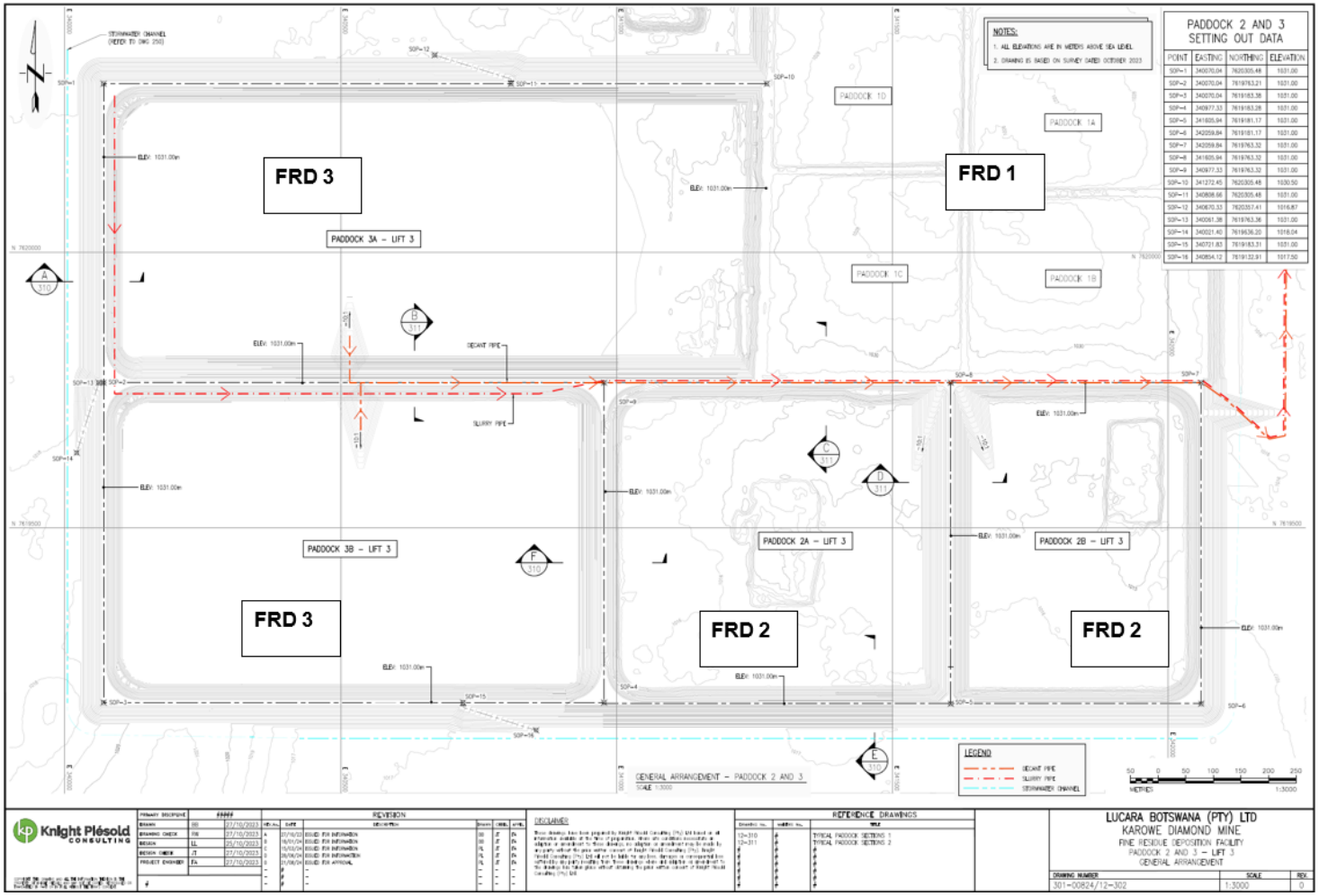
Closure considerations for FRD 3 are guided by the GISTM and informed by risks previously identified for FRD 1, including slope instability, groundwater contamination, climate change impacts on revegetation, dust emissions, and uncontrolled land use (Knight Piésold, 2024).

The closure objectives are to ensure long-term physical stability, control contamination risks, establish sustainable vegetation, and implement effective post-closure monitoring. Key activities include reprofiling side slopes to stable gradients, applying layered cover systems with growth media, establishing indigenous vegetation, managing stormwater through drainage and erosion control structures, and dismantling stormwater infrastructure (Knight Piésold, 2024).

Community engagement should be maintained throughout the closure process to align the outcomes with stakeholder expectations. Post-closure monitoring should address slope stability, water quality, and vegetation success, supported by routine inspections and maintenance programs (Knight Piésold, 2024).

Overall, the recommended closure strategy for FRD 3 aims to deliver a safe, stable, and rehabilitated landform that meets regulatory requirements, supports ecological restoration, and ensures long-term environmental protection in line with GISTM and ICMM good practice (Knight Piésold, 2024).

Figure 18-13: FRD 3 Proposed Layout Drawing



Source: KP (2024)

18.4.6 FRD Capacity Analysis

The final storage capacities for FRD 2 and FRD 3 were updated using historical deposition data, 2025 monitoring surveys, MUK3D modelling software and projected tonnages from the mine. In addition to the storage capacities, the required time to deposit the calculated volumes was also determined. The results are presented in Table 18- and Table 18-.

Table 18-8: FRD 2 Capacity Volumes and Time to Fill

Paddock	Planned Berm Elevation (mL)	Height Above Natural Ground Level (m)	Volume achieved (from 2025 (m^3)	Months	Years	Total Years
Raise 1						
Paddock 2A	1021	5 (current)	825,200	10.2	0.9	1.0
Paddock 2B	1021	5	171,400	2.1	0.2	
Raise 2						
Paddock 2A	1023	7	619,300	7.7	0.6	1.0
Paddock 2B	1023	7 (current)	374,500	4.6	0.4	
Raise 3						
Paddock 2A	1026	10	969,700	12.0	1.0	1.7
Paddock 2B	1026	10	668,800	8.3	0.7	
Raise 4						
Paddock 2A	1031	15	1,427,300	17.7	1.5	2.6
Paddock 2B	1031	15	1,110,900	13.0	1.1	
		Total	**6,110,100**			**6**

Notes:
Rate of rise = 1.5 m/year
Source: KP (2025)

Table 18-9: FRD 3 Capacity volumes and Time to Fill

Paddock	Planned Crest Elevation (mL)	Height Above Natural Ground Level (m)	Volume Achieved (m^3)	Months	Years	Total Years
Raise 1						
Paddock 3A	1019	3	68,300	0.8	0.1	0.5
Paddock 3B	1019	3	443,700	5.5	0.5	
Raise 2						
Paddock 3A	1021	5	526,700	6.5	0.5	1.5
Paddock 3B	1021	5	923,900	11.4	1.0	
Raise 3						
Paddock 3A	1026	5	2,222,800	27.5	2.3	5.1
Paddock 3B	1026	5	2,674,700	33.1	2.8	
Raise 4						
Paddock 3A	1031	5	2,033,200	25.2	2.1	4.3
Paddock 3B	1031	5	2,156,700	26.7	2.2	
		Total	**11,049,900**			**12**

Notes:
Rate of rise = 1.3 m/year
Source: KP (2025)

The original 2020 design for FRD 2 specified beach slopes of 1V:200H, with deposition planned on only two sides of the FRD. However, analysis of the July 2025 monitoring survey indicates that the achieved beach slopes average between 1V:120H and 1V:140H. This steeper slope has increased the Rate of Rise (RoR) around the embankment area (head of beach) with deep supernatant pond depths, impacting the facility's overall capacity. Analysis of the in-situ dry density results indicates a large variability between deposition months across 2023 and 2024. The current design allows for this variance with the additional capacity if required.

The revised capacity analysis accounts for these deviations. The QP recommends reassessing slurry velocity (Knight Piésold, 2024) and adjusting deposition techniques (deposition lines which circumnavigate more of the FRD embankment) to ensure full utilization of the FRD's capacity. Table 18-10 below shows various facilities development options to meet the LOM production targets and the capacity achieved from each option as of September 2025.

Table 18-10: FRDs Development Options for LOM

Facility	Proposed Final Facility Crest Elevation (mL)	Required LOM Capacity (Mm³)	Achieved LOM Capacity (Mm³)	Storage Capacity Utilization (%)
FRD 2	1026	11.1	14.7	75
FRD 3	1031			
FRD 2	1031	11.1	13.0	85
FRD 3	1026			
FRD 2	1031	11.1	17.2	64
FRD 3	1031			

Source: KP (2025)

As shown in Table 18-10, the updated LOM and storage capacity requirements have affected the projected utilization of the FRDs. This creates an opportunity to optimise the FRD footprints to improve capacity utilization and, in doing so, reduce overall capital costs.

18.4.7 FRDs and CRD Classification

18.4.7.1 Existing Facilities

The existing facilities (FRD 1,2 and CRD) were rated using the GISTM consequence category. According to the 2025 monitoring report (Geoflux, 2025), the facilities classify as follows.

Table 18-11: Recommended GISTM Consequence Category

Facility	Status	GISTM Classification
Coarse Residue Deposit		
Leg 1	Dormant	High
Leg 2	Active	
Fine Residue Deposit (FRD 1)		
Paddock A	Dormant	Very High
Paddock B	Dormant	
Paddock C	Dormant	
Paddock D	Dormant	

Facility	Status	GISTM Classification
Fine Residue Deposit (FRD 2)		
Paddock A	Active	High
Paddock B	Active	

Source: Geoflux (2025)

18.4.7.2 FRD 3

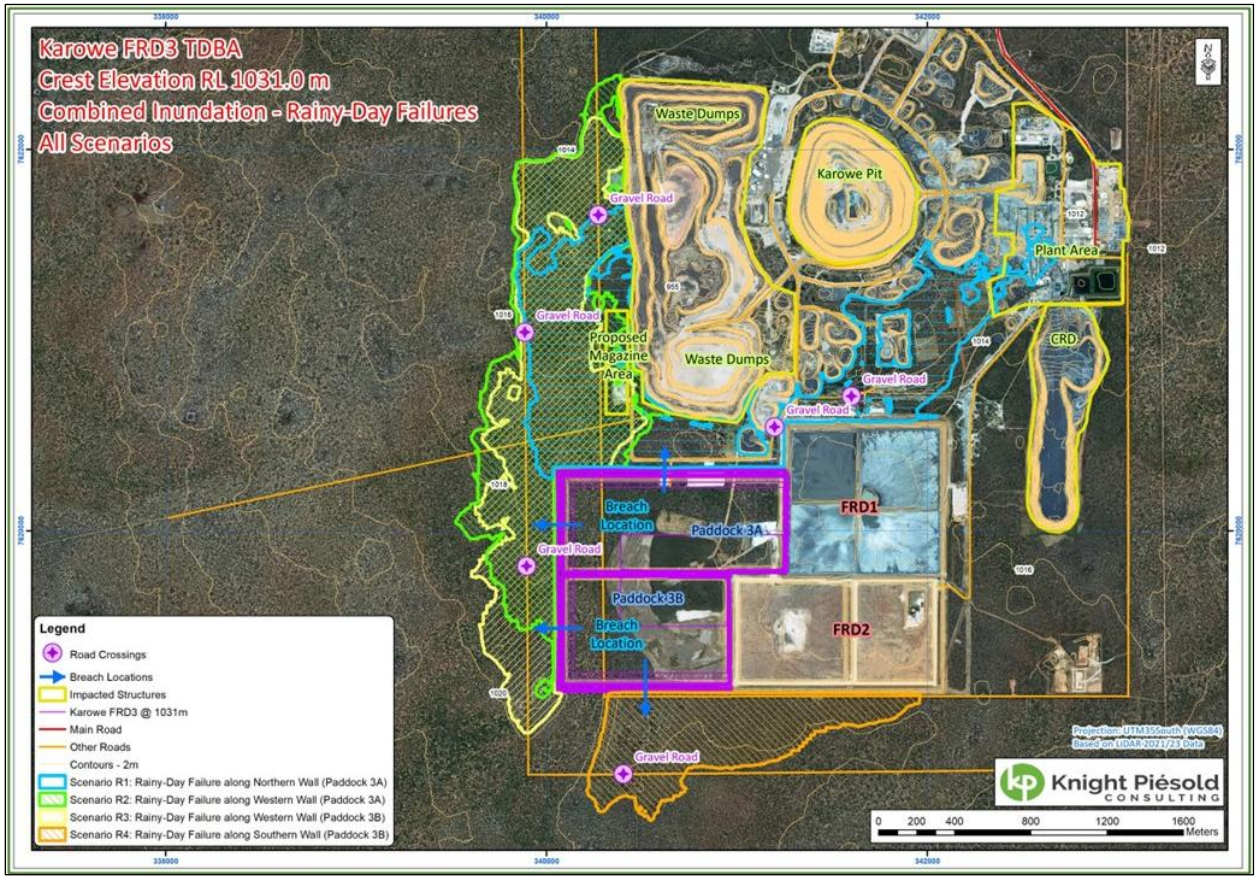
As part of the TDBA study, eight failure scenarios were evaluated for FRD 3 under both sunny-day and rainy-day conditions. The analysis considered potential breach positions and assessed the resulting extent of inundation downstream.

- Rainy-day scenarios focused on two mechanisms: (i) initial flood wave from pond water discharge, and (ii) mudflow/flow slide from liquefied tailings; and
- Sunny-day scenarios considered release of liquefied tailings due to loss of confinement.

The most critical case was identified on the northern flank under rainy-day conditions, with an estimated Potential Population at Risk (PAR) of 174 persons and Potential Loss of Life (PLL) of 0–3 persons. Breaches to the west would inundate the proposed magazine area and waste rock dump north of FRD 1.

Based on these outcomes, FRD 3 was classified as having a “High” consequence rating under the GISTM.

Figure 18-14: Consolidated view of the rainy-day failure scenario inundation boundaries



Source: KP (2024)

18.4.8 Stability Assessment

18.4.8.1 FRD 1

A two-dimensional Limit Equilibrium Analysis (LEA) was undertaken using Rocscience Slide 2 software, applying the Morgenstern-Price method of slices to evaluate potential slip surfaces and determine the minimum Factors of Safety (FoS) (Rocscience, 2024). Input parameters were derived from site surveys, previous investigations, and CPTu analyses, with phreatic levels estimated/validated using Finite Element seepage modelling. Analyses covered drained/static, undrained (peak and residual), and pseudo-static seismic loading conditions, benchmarked against ANCOLD (2019) and ICOLD (2022) best-practice FoS criteria.

The Independent Tailings Review (2023) recommended assessment of three failure modes: shallow localised sloughing, slope failure not intercepting tailings, and global failure intercepting tailings. The results indicate that the facility meets or exceeds recommended FoS values under

drained and undrained conditions, with post-seismic factors remaining within acceptable screening-level thresholds. These outcomes confirm that the facility demonstrates adequate stability under both operational and extreme loading scenarios.

Table 18-12: Recommended Factors of Safety (ANCOLD, 2019)

Loading Condition	Recommended Minimum for Tailings Dams	Minimum FoS Required
Long-term drained	1.5	Effective Strength
Short-term undrained (potential loss of containment)	1.5	Consolidated Undrained Strength
Short-term undrained (no potential loss of containment)	1.3	Consolidated Undrained Strength
Post-seismic	1.0 – 1.2	Post Seismic Shear Strength

Source: KP (2025)

Table 18-13: Recommended Factors of Safety (ICOLD, 2022)

Loading Condition	Recommended Minimum for Tailings Dams
Static Condition	1.5
Peak Undrained Strength	1.5
Residual Undrained Strength	1.1

Source: KP (2025)

The results of the slope stability analysis for FRD 1 on the three failure modes are summarised below for the critical section. Refer to Figure 18-16 for a location of each section analysed.

Table 18-14: Stability Results for FRD 1 - Shallow Localised Sloughing

Section	Drained/Static	Drained Pseudo-Static	Undrained Peak Ratio	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.1	1.5	1.1
	Achieved FoS			
SDA 1	1.53	1.28	1.54	1.54

Notes:
Green: Acceptable
Orange: Marginal
Red: Unacceptable
Source: KP (2024)

Table 18-15: Stability Results for FRD 1 - Slope Failure

Section	Drained/Static	Drained Pseudo-Static	Undrained Peak Ratio	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.1	1.5	1.1
	Achieved FoS			
SDA 1	1.67	1.39	1.67	1.67

Notes:
Green: Acceptable
Orange: Marginal
Red: Unacceptable
Source: KP (2024)

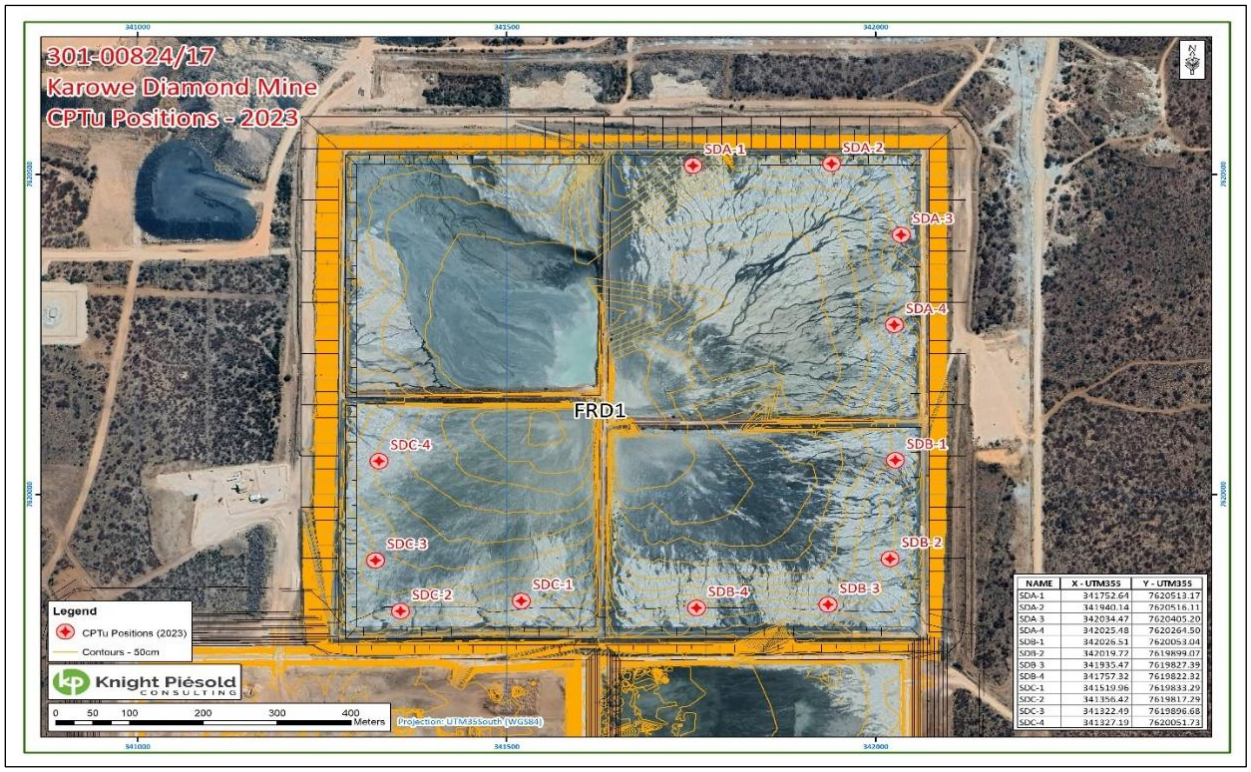
Table 18-16: Stability Results for FRD 1 - Global Failure

Section	Drained/Static	Drained Pseudo-Static	Undrained Peak Ratio	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.1	1.5	1.1
	Achieved FoS			
SDA 1	1.85	1.51	1.84	1.85

Notes:
Green: Acceptable
Orange: Marginal
Red: Unacceptable
Source: KP (2024)

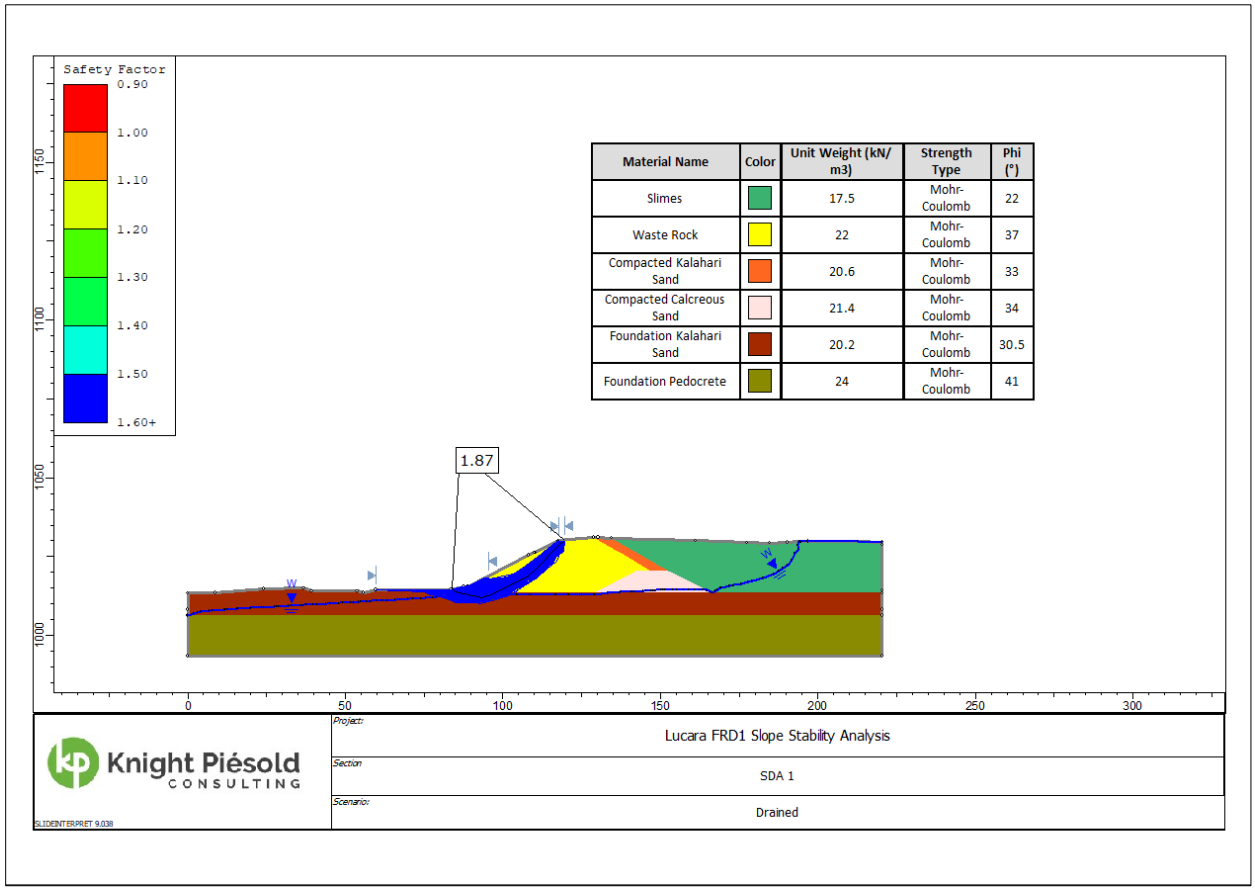
Figure 18-15 shows the positions of the probes on FRD 1. A typical section illustrating the drained scenario for Global Failure has been shown on Figure 18-16.

Figure 18-15: CPTu Probe Holes



Source: KP (2024)

Figure 18-16: Global Failure Slip Surface for Section SDC A Drained



Notes:
Section view is looking East
Source: KP (2024)

The slip surface illustrated in the figure above represents the minimum global factor of safety (FoS) associated with failure surfaces that intersect only the engineered embankment of the facility. Potential slip surfaces that extend into the tailings exhibited significantly higher FoS values and were therefore not considered representative of a conservative minimum-FoS condition for the facility.

18.4.8.2 FRD 2

The stability of FRD 2 was evaluated using GeoStudio's SEEP/W and SLOPE/W modules, with geometry derived from a 2020 LiDAR survey and modelling on Muk3D. Material properties were sourced from previous geotechnical investigations; Sands Civils Services (SCS) (2019), the 2019 Feasibility Study of Karowe Diamond Mine (FRD & CRD) (Knight Piesold, 2019). Phreatic surfaces were modelled in SEEP/W and imported into SLOPE/W, with analyses run for drained, undrained (peak and residual), pseudo-static seismic, and upset conditions (blocked toe drain).

Seismic loading was assessed using site-specific hazard values, with coefficients ranging from 0.08 g to 0.16 g (corresponding to AEPs of 1:500–1:10,000). Contractive behaviour was assumed for slimes, while waste rock and other compacted materials were modelled as dilative. Factors of Safety were benchmarked against ANCOLD (2012) best-practice values as shown in Table 18-17.

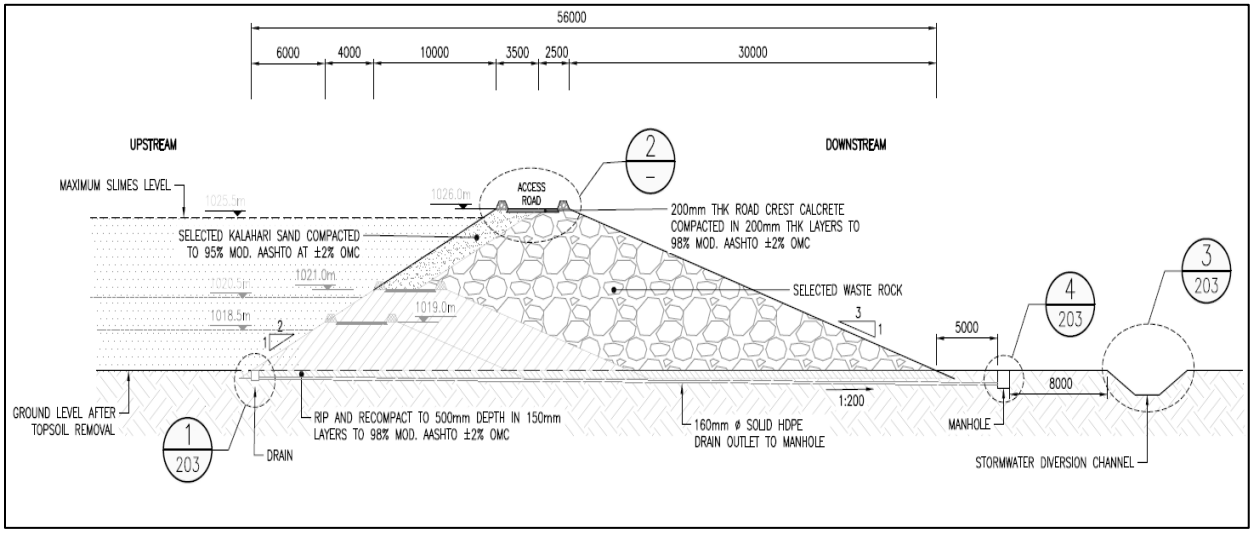
Table 18-17: Minimum FoS as required by ANCOLD

Parameter	Minimum Required
Drained	1.5
Undrained Peak ratio	1.3
Undrained Residual ratio	1.1
Drained Pseudo-Static	1.1

Source: KP (2025)

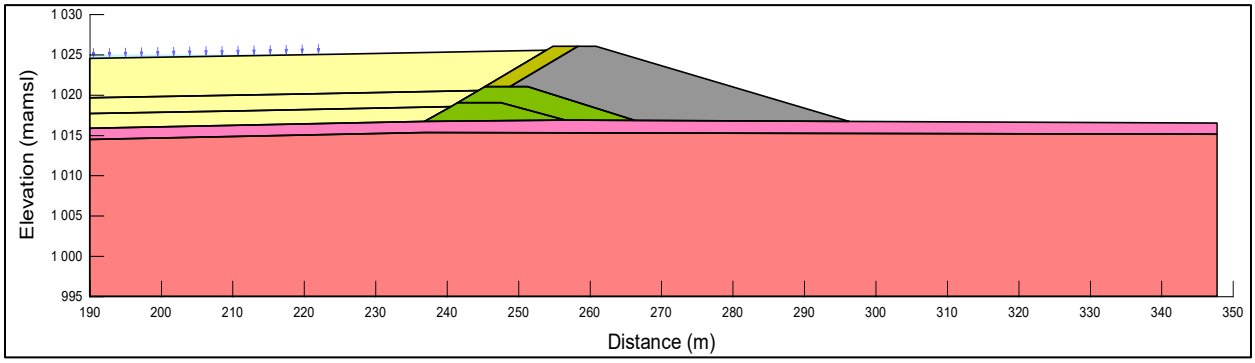
Foundation stratigraphy was based on geotechnical investigations by SCS (2019–2020) and confirmed in this study. The results of the analysis demonstrated that FRD 2 achieved or exceeded the required FoS under the evaluated loading conditions, confirming compliance with best practice and adequate stability of the design. These assumptions, together with other design aspects, are as presented in the typical drawing cross-section below.

Figure 18-17: Typical embankment section Detail



Source: KP (2022)

Figure 18-18: Typical Section Showing Lifting Sequence



Source: KP (2022)

The results of the slope stability analysis for the proposed FRD 2 facility are summarised in Table 18-18.

Table 18-18: FRD 2 Stability Results

FRD lifting sequence		Drained	Undrained Peak Ratio	Undrained Residual Ratio	Drained Pseudo-Static
		Minimum Required (Global Failure)			
		>1.5	>1.3	>1.1	>1.1
Lift number	Lift 1.1	>1.5	>1.3	>1.1	>1.1
	Lift 1.2	>1.5	>1.3	>1.1	=1.1
	Lift 2	>1.5	>1.3	>1.1	=1.1

Source: KP (2022)

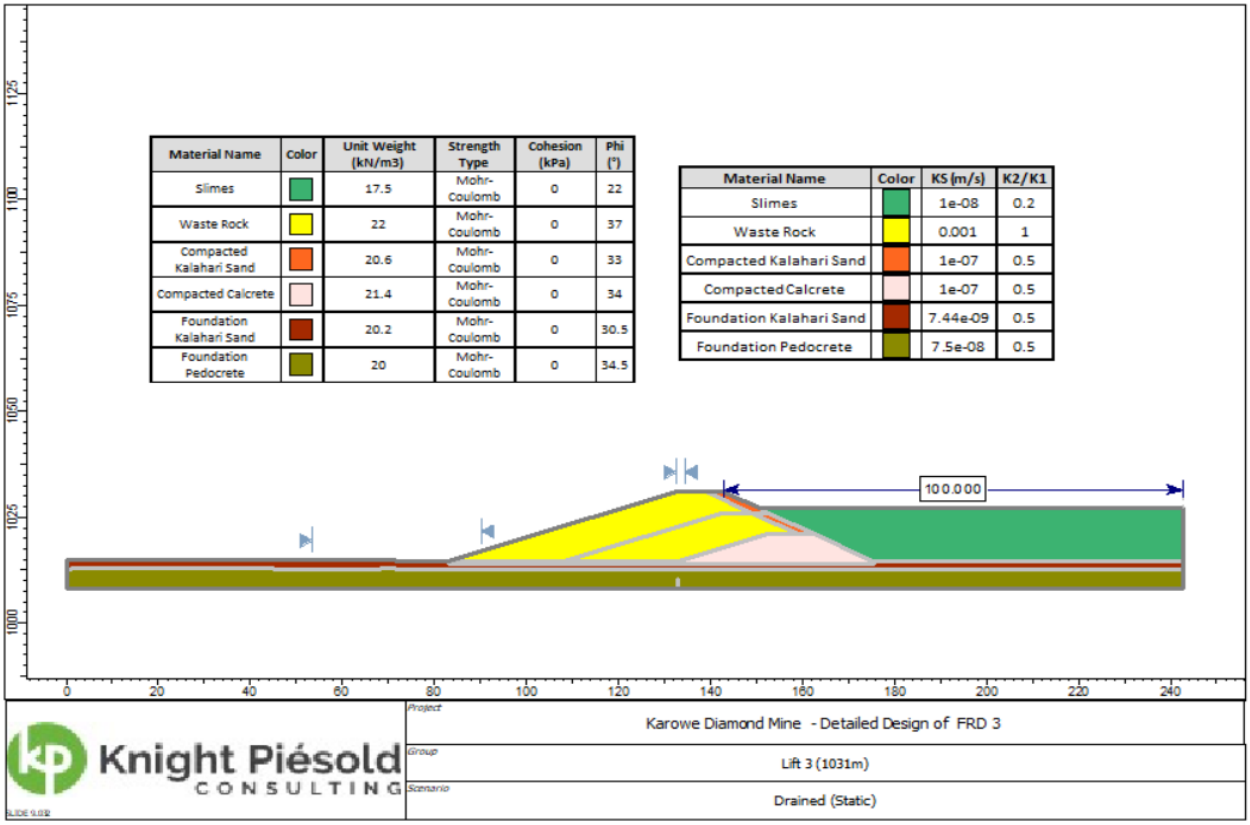
18.4.8.3 FRD 3

The critical section for FRD 3 was used for this seepage and stability analysis. The layering or stratigraphy of the critical section was determined based on rotary core borehole, BHK09, located near the section. BHK09 revealed the presence of Aeolian (silty sand) deposits extending to 2 metres below ground level (mbgl). Beneath the sand layer was a pedocrete (calcareous sands) layer extending to a depth of 7.5 m. Below the pedocrete, a 1.5 m thick layer of silcrete was identified.

It is important to note that the thickness of the Kalahari sand along the walls of FRD 3 extends to a depth of 4 to 5 mbgl, as indicated by borehole tests. However, on the northeast wall where FRD 3 abuts FRD 1, the Kalahari sand is only 2 m thick.

The location and the cross-section of the critical section for FRD 3 are shown in the figures below.

Figure 18-19: FRD 3 Critical Section Stability Model (at 1,031 mamsl)



Source: KP (2024)

As per the Independent Tailings Review performed in November 2023 (Grant-Stuart, Brown, & Brink, 2023), it was recommended that the stability analyses should consider and report three failure modes:

- Shallow localised sloughing;
- Slope failure (not intercepting fine tailings); and
- Global failure intercepting tailings.

The results of the slope stability analysis for FRD 3 on the three failure modes are summarised in Table 18-19 through Table 18-.

Table 18-19: Stability Results for FRD 3 – Sloughing/ Localised Failure

FRD Lifting Sequence	Drained/Static	Undrained Peak Ratio	Undrained Pseudo-static	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.5	1.1	1.1
Lift 1 (1021 m)	2.10	2.10	1.66	2.10
Lift 2 (1026 m)	2.24	2.24	1.76	2.24
Lift 3 (1031 m)	2.22	2.22	1.74	2.22

Source: KP (2024)

Table 18-20: Stability Results for FRD 3 – Slope Failure

FRD Lifting Sequence	Drained/Static	Undrained Peak Ratio	Undrained Pseudo-static	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.5	1.1	1.1
Lift 1 (1021 m)	2.28	2.28	1.77	2.28
Lift 2 (1026 m)	2.30	2.31	1.80	2.31
Lift 3 (1031 m)	2.27	2.26	1.77	2.26

Source: KP (2024)

Table 18-21: Stability Results for FRD 3 – Global Failure

FRD Lifting Sequence	Drained/Static	Undrained Peak Ratio	Undrained Pseudo-static	Undrained Residual Ratio
	Minimum Required FoS			
	1.5	1.5	1.1	1.1
Lift 1 (1021 m)	2.84	2.81	2.10	2.81
Lift 2 (1026 m)	2.56	2.58	1.97	2.58
Lift 3 (1031 m)	2.90	2.90	2.15	2.90

Source: KP (2024)

The Factors of Safety (FoS) demonstrates that both drained and undrained conditions of the facility meet the stability criteria as recommended by the International Commission on Large Dams (ICOLD) for all three failure modes. This stability is largely due to the presence of a filter drain on the upstream side of the calcrete starter wall.

The water table is low within the region. The FRD 3 seepage and stability model remains conservative as the water table is set at 4 mbgl despite the lack of groundwater seepage near the existing and active FRD's, and the relative permeability differences between the slimes and the foundation material. Furthermore, the use of an undrained peak and residual ratio of 0.26 for the Kalahari sand is considered conservative, especially given that the consolidated undrained triaxial tests indicate strain-hardening behaviour in this material (Knight Piésold, 2024).

18.4.9 Storm Water Management

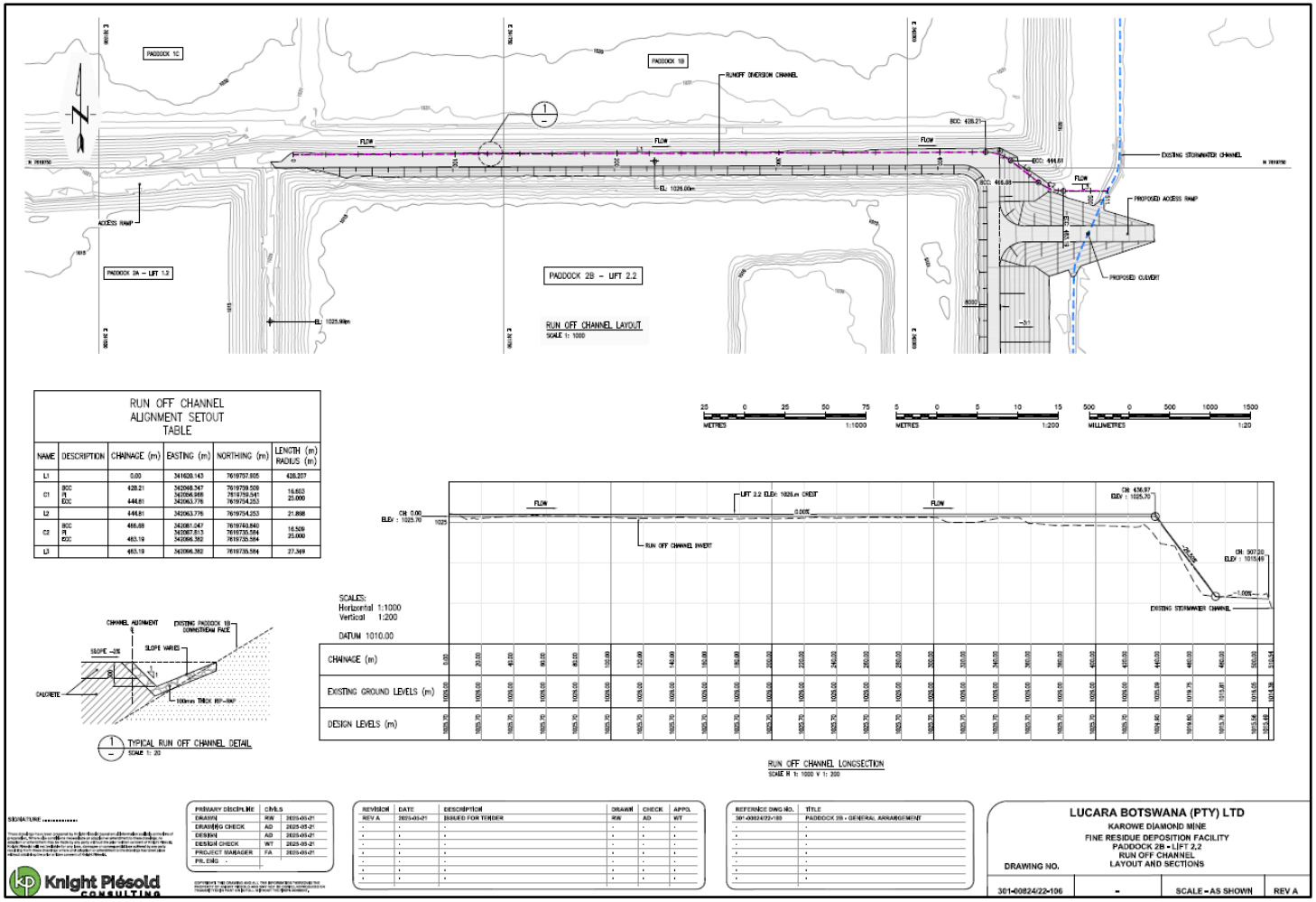
18.4.9.1 FRD 2

A stormwater management design and plan was developed as part of the detailed design for FRD 2 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 (South African standard). This involved 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).

Contact water generated from the embankment walls of the FRDs is diverted away from the site perimeter using a stormwater channel (Knight Piésold, 2022). The stormwater channel is an unlined earth canal that conveys contact runoff along the perimeter of the FRDs, before discharging to the environment away from the FRDs. The channel falls within the natural ground slope of the land to avoid excessive excavation, routing it to a controlled discharge point.

In the 2025 design for the FRD 2 lift, implementing a controlled drainage system was considered essential to manage runoff from the higher FRD 1 embankment slope, which was causing erosion on the FRD 2 crest. Consequently, a runoff channel was integrated into the design, as illustrated in the figure below, to effectively divert runoff into the existing stormwater channel.

Figure 18-20: FRD 2 Paddock B Stormwater Management



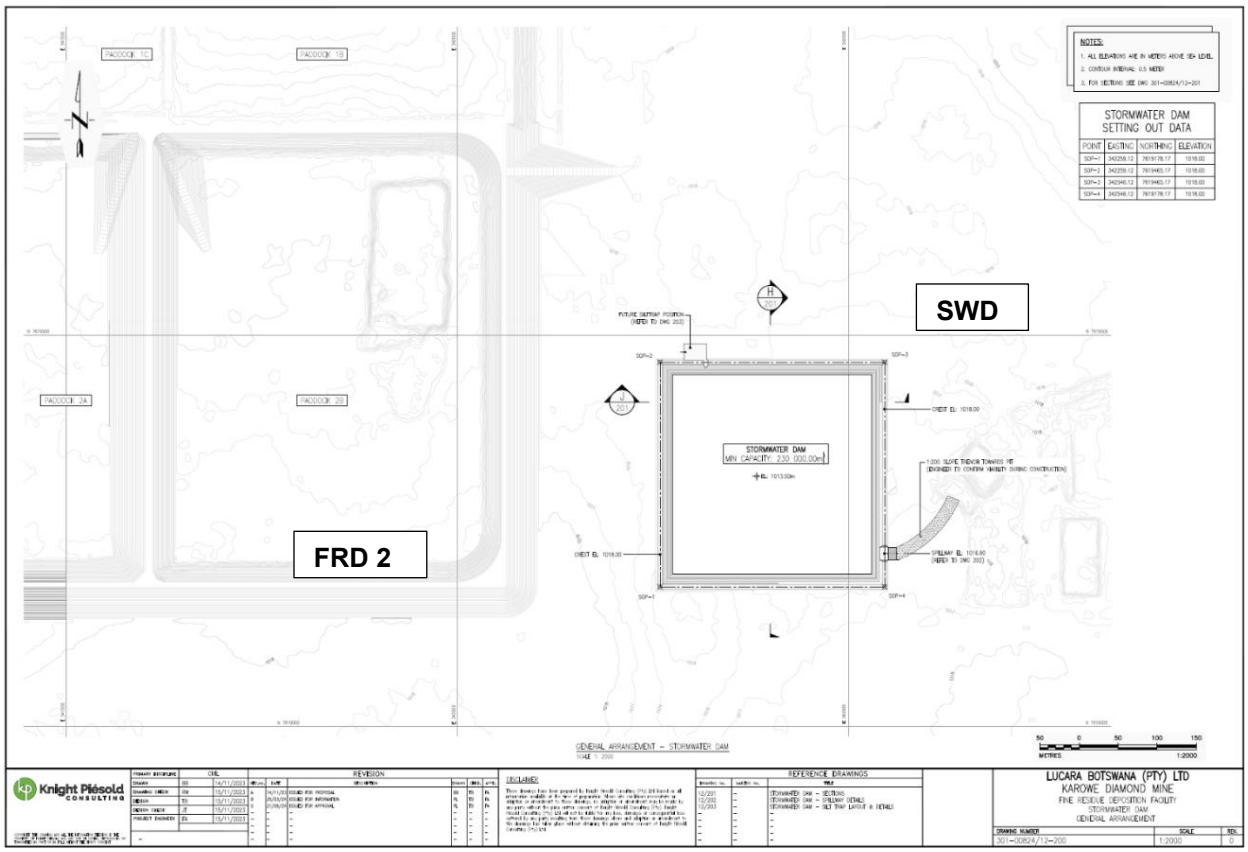
Source: KP (2025)

18.4.9.2 FRD 3

During the FRD 3 design, a full stormwater management plan was developed to effectively manage surface water runoff, prevent flooding, and ensure the safe operation of the mine. By using site specific rainfall data and hydrological analysis, the design accommodates storm events and mitigates risks associated with water flow, erosion, and sediment build-up.

This infrastructure is optimised to manage stormwater runoff and prevent flooding, requiring regular maintenance to ensure its long-term efficiency. The figure below shows the stormwater dam layout.

Figure 18-21: Proposed SWD Layout



Source: KP (2024)

18.4.10 Water Balance

The Karowe Diamond Mine water management system integrates a single processing plant, three Fine Residue Deposits (FRDs), a Process Water Dam (PWD), an evaporation pond (EVAP), and a planned Stormwater Dam (SWD). The system is designed to maximise water return to the plant, reduce groundwater losses, and minimise the need for additional extraction (Knight Piésold, 2024).

The following conclusions were made from the 2024 FRD 3 Feasibility Study:

- The available climate and rainfall data from the neighbouring towns were analysed to describe the baseline conditions on the site. The rainfall from the Letlhakane Weather Station was chosen, since the length and the reliability of the recorded data is promising and its proximity to site. Daily rainfall and average monthly evaporation records were produced for use in the determination of the water balance. In addition, the temperature statistics were available for the Letlhakane station and thus reported;

- The water balance recommended for a SWD with a capacity 146,664 m³. The dam was sized based on allowing water to back up onto the FRDs during large storms to reduce pressure on the dams but not to the point where instability could be induced. The backed-up water will be removed within a period which limits seepage and elevated phreatic levels The size of the dam can be adjusted along with the water being backed up onto the FRDs to achieve an optimal size of the dam should it be required;

- The water balance was run on the predication that the new stormwater dam be built and operated in such a way that the decant and seepage from the FRDs can be directed to the SWD when the PWD reaches 90% of its capacity. This will allow both dams to operate without unnecessary spillages and maximise the reuse of process water within the mine circuit;

- While the spillage statistics mention the PWD spilling at a return period of 1 in 8 years this is inconsequential as it spills into the adjacent evaporation pond which has an external spill return period of 150 years far beyond that of the 50-year spill criteria required; and

- The proposed SWD will have a high return period for spillage at 156 years.

18.4.11 Conclusion

The feasibility designs have been guided by the level of investigation and analysis consistent with the GISTM and international practice requirements. Geotechnical and hydrogeological characterisation has included ERT & MASW geophysics, rotary core drilling, test pits, construction and test-pumping of monitoring boreholes, seismic hazard assessment, and development of groundwater flow models. These datasets have been integrated into stability, seepage and hydrogeological assessments to confirm foundation suitability, demonstrate controlled pore-pressure regimes, and support construction and operational concepts that are aligned with the GISTM.

The tailings management facilities designs are robust and stable facilities that provide sufficient storage capacity to meet the current LOM requirements. The embankment geometries, drainage,

and deposition strategies have been developed to achieve appropriate factors of safety under both static and post liquefaction scenarios to comply with ICOLD & ANCOLD requirements. A dedicated stormwater dam and associated surface-water controls have been incorporated to keep the site operable during major storm events and to minimise rain-induced instability in accordance with GN 704 (South African standard). Overall, the tailings facilities and associated water management systems constitute a fit-for-purpose tailings management solution consistent with current global expectations for safe and responsible tailings practice.

19 MARKET STUDIES AND CONTRACTS

19.1 Lucara Diamond Marketing

This section is contributed by Lucara under the oversight of independent QP Dr. Hermanus Grütter. The information documented herein is updated where relevant to the end of September 2025.

Under the terms and conditions contained within the Mining Licence (ML 2008/6L), Lucara Botswana will hold any open tenders for diamond sales in Botswana and may, with approval of the Government of Botswana, offer open tender sales internationally. In the period 2012 to the end of 2014, dual viewings of goods were held in Antwerp and Gaborone with the final tender closing in Antwerp. From January 2015 to 2020, all diamond tender viewings and sales took place in Lucara's dedicated sales and marketing office in the Diamond Technology Park in Gaborone, Botswana. In 2020, during the Covid-19 pandemic, Lucara received permission to once again hold dual viewings of tender goods in Gaborone and Antwerp with final tender closing in Antwerp and, as of September 2025, this mechanism was still in effect.

A portion of the KDM production, mainly diamonds in the range from 1.5 ct to 10.8 ct in the better colours, shapes and clarities have been sold through the Clara Diamond Solutions platform (Clara) since 2018. The Clara platform matches buyers to sellers on a stone-by-stone basis based on polished outcome. Clara has a client base of >100 customers and sells diamonds on average every 1 to 2 months depending on supply. Approximately 2% by weight (and 14% by revenue) of the KDM production is currently sold on the Clara platform.

KDM's large, high value +10.8 ct 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. 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. Lucara terminated the agreement in September 2023 due to financial breach.

A New Diamond Sales Agreement (NDSA) for a ten-year term, was entered into in February 2024, effective December 2023. Under the NDSA, +10.8 ct gem and near gem diamonds from KDM of qualities that can directly enter the manufacturing stream are 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 the HB marketing and polishing margins. If the final sales price is higher than the initial estimated polished price, a top-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 marketing and polishing margins) will result in the difference being refunded to HB. Top-up payments 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.

Ahead of the expiration of the HB Antwerp New Diamond Sales Arrangement (NDSA) in 2033, the Company intends to evaluate and implement alternative sales strategies consistent with prevailing market conditions and the requirements of major luxury brands. These strategies are to reflect the Karowe Diamond Mine's (KDM) unique production profile, including an increased proportion of large diamond recoveries and a corresponding expansion of the product range, realizing a projected 10.5% increase in diamond price assumptions.

The revised sales framework is designed to optimise the Company's marketing and sales approach, reduce reliance on third-party execution, enhance price discovery through broader buyer engagement, and provide increased flexibility with respect to sales timing and channels. These measures are expected to preserve realized prices while mitigating certain manufacturing, distribution, and financing risks. All +10.8 ct non-gem quality diamonds and all diamonds less than 10.8 ct in weight that do not meet the criteria for sale on Clara are sold as rough through a quarterly tender. Lucara manages a rough price book 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 NDSA and reserve prices are compared prior to release. The costs of the GDV are for the account of the Government of the Republic of Botswana (GRB). Royalty payments are calculated on the actual sales price achieved during tenders, sales through the Clara platform and final polished value sold under the HB Antwerp NDSA.

19.2 Diamond Sales

Since 2012 until the end of September 2025 over 5 Mct of combined North, Centre and South lobe diamonds have been sold for revenue of US$2.7 B (average price per carat of US$545/ct).

Sales lots are prepared by Lucara Botswana staff for presentation to tender clients, supply to Clara and HB Antwerp in a modern, ultra-secure sorting facility. Tender sales 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 using a secured online platform. Results are announced at the close of the tender and witnessed by a court appointed 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 licence, and compliance to the Kimberley Process.

Clara sales are generally held every 1-2 months and are 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 clients' 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 US$1.0 million per diamond. Lucara has discontinued selling through EST's and in 2020 established an offtake agreement with HB Antwerp, with a 10-year extension signed in 2024.

19.3 Client Base

Lucara has developed a strong, geographically diverse client base. Lucara has over 700 registered clients, demonstrating a strong interest in the KDM production for those goods sold via tender. Lucara has an offtake agreement with HB Antwerp for all +10.8 ct diamonds that are suitable for polishing.

19.4 Rough Diamond Market Outlook

The overall rough and polished markets have experienced significant fluctuations since 2019, with diamond pricing improving toward the latter part of 2019, followed by a sharp and deep decline in pricing during the first three quarters of 2020 due to the global pandemic. During 2022 there was a marked increase in rough diamond prices — the quantum of value increase exceeded the pandemic-low and held firm over a longer period. During 2023 rough pricing again declined sharply toward pre-pandemic levels and in certain categories to values observed during the pandemic-low. As a result of companies decreasing volumes of goods for sale and India imposing (or considering) import restrictions on rough, diamond prices began to show signs of rebound in late 2023 and into early 2024. Increasing market share from lab-grown diamonds continued to exert downward pressure on smaller-sized natural rough.

Negative rough-pricing pressure combined with upward inflationary pressure across the supply chain since 2019 resulted in some mine closures and companies seeking creditor protection (including Firestone, Lucapa, Burgundy and Stornoway Diamonds), as well as curtailment or deferral of capital expenditure at mines such as Cullinan and Finsch (Petra Diamonds), Gahcho Kué (Mountain Province Diamonds) and Letšeng (Gem Diamonds).

Issues that during 2022-2025 continue to apply downward pressure on the rough diamond market include:

- The Ukraine/Russia conflict, with supply of Russian goods largely channeled through non-G7 countries and causing trade/tariff fragmentation;
- Lab-grown diamonds impacting smaller sizes of rough in the lower qualities, combined with negative commentary regarding natural diamonds;
- US import tariffs (and other trade/tariff risks) on specific diamond-producing and beneficiation countries;
- Lower demand from major markets such as the USA, and sluggish demand recovery in China; and

- Elevated inventories in manufacturing hubs (notably India) and supply-chain imbalances in smaller goods.

The longer-term outlook for natural diamond prices remains cautiously positive, anchored on improving fundamentals around supply and demand as many of the world's largest mines approach their natural end-of-life. Following on from the record-high diamond prices achieved in early 2022, a softer diamond market emerged in the latter half of 2022 which has persisted into 2024 and into mid-2025 — the result of global economic concerns, geopolitical uncertainty, and inventory overhang. Prices continue to show signs of stabilisation, although recovery remains modest and selective. The growth of lab-grown diamonds remains an ongoing structural challenge for the lower end of the natural-rough market, although natural diamonds remain differentiated by rarity and brand prestige.

For Lucara, a strong and expanding customer base, excellent participation in tenders and the exposure to polished-sale upside (via the HB agreement on large, high-value Type IIa stones) combined with a consistent production profile incorporating higher-grade South Lobe / EM/PK(S) production, positions the Lucara brand toward a diamond-price outlook that is relatively more positive. Exposure to market fluctuations for the Karowe mine production can be expected, but 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, 2024; or (ii) prior to January 1, 2024 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 (WCF) are being made available to Lucara Botswana by way of a senior secured term loan facility in the principal amount of up to US$170 million and a senior secured revolving credit facility in the principal amount of up to US$50 million. On June 9, 2024, the Facilities Agreement was amended to a senior secured term loan facility in the principal amount of up to US$190 million and a senior secured revolving credit facility in the principal amount of up to US$30 million. 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 US$190 million from the Project Loan and US$30 million from the WCF. The cost overrun reserve account (CORA) was fully funded in June 2025 with US$28 million subsequently being drawn from the account. The balance in the CORA stands at US$33.7 million on September 30, 2025.

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.

Outlined here are key terms of the Facilities Agreement, as amended:

- Up to US$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;
- Eight (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 US$61.7 million was fully funded in accordance to the agreement and the Company has drawn US$28 million subsequently to fund UGP expenditure;
- 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 Working Capital Facility (WCF):

- Up to US$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 WCF on the Available Commitment for the WCF.

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 US$63.0 million 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. 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 US$500,000 drawn, calculated monthly, should any amount be drawn under the Shareholder Guarantees, subject to TSX and other regulatory approvals (the Nemesia Consideration). As of September 30, 2025, US$10 million has been drawn from the shareholder standby undertaking.

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 significant bonus and penalty components which incentivizes JDS to achieve project cost and project schedule targets. On March 31, 2025, the EMPC contract with JDS Foreign Enterprises Inc. was terminated in favour of a Master Service level Agreement; 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 non-material 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 reporting, two pre-mining environmental studies were conducted for KDM (formerly known as the AK6 Project), namely an EIA Study for AK6 (Geoflux, 2007) and 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 Licence (ML 2008/6L); Boteti Mining was granted common law surface rights over the entire mining licence 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 Licence (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 licences and permits are presented in Table 20-3 below.

Regulatory approval was granted in August 2024 for the on-site storage and mechanical evaporation of saline groundwater (TDS ±30,000 mg/l) in a lined pond. By 2031, additional produced water disposal plans will need to be developed for the remaining LOM. This future plan will be subject to an Environmental Impact Statement (EIS) and its regulatory approval.

20.1.3 Capacity

The Environment, Health, Safety & Community Relations (EHS & CR) Department comprises approximately 37 positions (including one fire officer, one fire captain, and four firemen). The department includes dedicated health and safety, medical/wellness, sustainability, environmental, waste management, stakeholder engagement as well as corporate social investment functions.

20.1.4 Environmental Management

With a producing mine, Lucara demonstrates that it follows Good International Industry Practice, as defined by the International Finance Corporation (IFC), and as evidenced below.

- Lucara is a certified Member of the Responsible Jewellery Council. Its Code of Practices (2019) Standards covers ethical, social, human rights and environmental practices, and conformance is reviewed periodically by independent auditors. Lucara's latest certification was in March 2024 and the re-certification is expected in March 2027;

- 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 for its occupational health and safety management system, and has aligned its environmental management system with ISO 14001. The last recertification was in November 2024. A surveillance audit was performed in September 2025 with no major findings;

- Lucara has conducted annual self-assessments of alignment with the Mining Association of Canada's (MAC) Towards Sustainable Mining (TSM) Standards since 2021, and two external verifications from the QP's MAC certified assessors (latest verification: December 2024, valid for a 3-year period ending December 2027). Protocols verified include Biodiversity Conservation Management, 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

- As detailed further below in Tailings Management, Lucara has adopted the Global Industry Standard on Tailings Management (GISTM) for Fine Residue Deposit (FRD) and Coarse Residue Deposit (CRD) facilities. Lucara has engaged with an independent Engineer of Record and the first review by a three-person Independent Technical Review Board was completed Q4 2023 and a subsequent review completed in Q4 2024, with another scheduled for Q4 2025.

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 on, 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:

- 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 analysed 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 characterised 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 temperatures 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 savanna and mopane shrub savannah with occasional grassy areas. Most of the surface flow tends to be localised 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 Licence (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 Licence 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, 2024).

Table 20-1: Species with Conservation Status at KDM and Transmission Line

Conservation Status	IUCN	Botswana Red Data Book	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 licenced area
Threatened	0	2	Devils claw (*Harpagophytum procumbens*) and Hoodia (*Hoodia currorii*) in the licence 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 licenced area

Source: Ecosurv (2021), Lucara (2023, 2024)

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.

In July 2024, KDM conducted a comprehensive Biodiversity Assessment and Survey with internal staff and assistance from Birdlife Botswana. The survey emphasized richness and composition of local plant and animal species, revealing KDM supports at least 22 tree species, 12 grass species and a minimum of 12 animal species, including hyena (*Hyaena brunnea*) and African leopard (*Panthera pardus*). Mammalian species distribution was determined to be affected by illegal hunting and variations in vegetation composition.

The 2024 MAC TSM external audit rated KDM as AAA in both commitment, accountability and communications, and conservation planning and reporting. The audit rated KDM as AA in biodiversity reporting, recommending better focus on Community of Interest (COI) engagement on the KDM biodiversity platform and reporting.

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 year 2024 indicates annual abstraction from pit dewatering and water supply boreholes in 2024 was 1,805,424 m^3.

Section 18.2 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 that are under consideration or have been implemented.

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 are installed for the

disposal of the saline water to be pumped to the surface. Completion of the water management pond (including lining of pond) took place during Q2 of 2025. The water blending circuit was also commissioned in Q2 2025. The blending circuit currently is used to mix high-salinity underground water with low-TDS shallow aquifer water (sourced from pit-dewatering) to reduce the concentration of dissolved solids to an acceptable level for operational use. The treated water is then to be transferred to Debswana's Orapa Diamond Mine under the terms of a Memorandum of Agreement for water supply. Section 18.2.4. provides a description of the planned mechanical evaporators/ pond system and of its operation.

A Hydrological Impact Assessment for both the UG dewatering and desalination pond was completed in May 2024. Additionally, an EMP for evaporators was completed in August 2024. The annual groundwater monitoring report was completed in May 2025.

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 Global Climate Models (GCM)/ Regional Climate Models (RCM) ensemble. By 2050, most countries 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.

Lucara's 2024 Sustainability Report summarises GHG emissions and intensities as tabulated below.

Table 20-2: Lucara's GHG Emissions and Intensities for 2020 to 2024

Greenhouse Gas Emissions	2020	2021	2022	2023	2024
Scope 1 (tCO_2e)	18,692	16,618	26,685	20,504	16,090
Scope 2 (tCO_2e)	55,443	58,494	59,116	67,600	80,871
GHG Intensity (Total CO_2e(kt)/ore + waste rock mined (t))	13.5	11.7	17.9	15.2	24.57

Source: Lucara (Sustainability Report 2024)

Lucara Botswana has developed a Decarbonization Strategy, generated a Decarbonization Action Plan, and contracted Mott MacDonald to conduct a prefeasibility study for a large-scale solar photovoltaic (PV) project for KDM. In 2024, Lucara initiated engagement with suppliers to assess the availability of data required for Scope 3 emissions reporting and in 2025, Lucara begun collecting data to evaluate the 15 categories established by the GHG Protocol.

In 2024, Lucara conducted a Climate Change Risk Assessment in conjunction with the Lundin Foundation and continues to investigate opportunities to create a solar farm within the lease area. Interested parties have been invited to submit proposals to Lucara to develop a solar farm from which Lucara could purchase electricity via a power purchase agreement.

The 2024 independent MAC TSM verification found a level AAA rating for corporate climate change management but noted Facility climate change management at level A and Facility performance targets and reporting at level B. Discussions with the Lucara Botswana Climate Change Committee included improving performance in setting targets and reporting.

20.1.9 Tailings Management

Section 18.4 Tailings Management Facility details the (a) CRD, which comprise coarse tailings placed in designated storage areas, without containment structures such as walls or dams, and (b) two FRD facilities, 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 of the facility are currently underway and a closure and rehabilitation plan was completed in 2024. 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 GISTM criteria (Lucara, 2023).

As detailed in Section 18.4 TDBA analysis based on 2022 design specifications of FRD 2 recommended a “High” Consequence Classification in terms of the GISTM classification system. The mine has undertaken a risk assessment exercise to re-evaluate the credible failure modes which preliminarily suggest the risk will be lower. The FRD 2 detailed design was completed in 2022. In 2026, the mine will conduct further TDBA which could eliminate the need for a deflector berm.

The detailed design of FRD 3 was completed in 2024. In 2024 the company implemented a Tailings Management System that is aligned with GISTM, MAC TSM Tailings Protocol, and Botswana’s Mines, Quarries, Works, and Machinery Act CAP 44:02. The company has had two site visits by an Independent Technical Review Board (ITRB) in 2023 and 2024 (Duncan et al., 2023, 2024), with another in scheduled for Q4 2025, and is working toward full GISTM compliance with no significant noncompliance findings from the ITRB.

The MAC TSM independent verification in 2024, which also included a review of ITRB documents, scored KDM as AAA in all criteria except operation and maintenance, which is expected to be resolved in the 2025 ITRB review and upgrade this criteria to AAA. The company

has conducted community and local stakeholder engagement with respect to tailings management and dam breach analysis.

20.1.10 Waste Rock Storage Facility

The Waste Rock Storage Facility (WRSF) is located west of the FRD dams 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 a hardened surface road. Letlhakane is a regional center in central Botswana with a number of diamond mines operating within 75 km to the west and northwest of the town.

According to the Central District Integrated Land Use Plan (CDILUP) (Geoflux, 2007), the primary use for tribal land in the Boteti 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, 2020) included a Social Impact Assessment (SIA) 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 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, including four primary schools, three junior secondary schools, 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 are associated with the mine's operations and expansion. Issues of concern are voiced during public consultation. The primary concerns for most stakeholders 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 the eventual closure of the mine. 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.

The revised Stakeholder Engagement Plan (SEP) (July 2025) incorporates new stakeholder and social data gathered through a Position Paper on Vulnerability among Remote Area Dwellers, as well as the Community Needs Assessment (Karowe Diamond Mine IESC Construction Monitoring Report 7, Ver 1.0), 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 US$0.4 million in 2024 and an additional US$8.2 million in the Sports Complex over several years, and support initiatives ranging from developing a sports facility, ameliorating malnutrition through support of farming cooperatives, and GBV awareness raising.

The MAC TSM independent audit in 2024 rated all seven categories of Indigenous and Community Relations to be AAA.

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 collaborated with Debswana to study influx in the Boteti region. The report was finalised in Q4 2023 and concluded that the area around Letlhakane was a specific zone where influx was occurring. A series of recommendations were made, and the implementation of these recommendations was suggested to be collaborative and independent for the participants in the study. Jointly Lucara, Debswana, and the Boteti District Council have developed the Boteti District Site-Induced Migration Management Plan (SIMMP), which was approved in October 2024. The Boteti District SIMMP (2024-2029) has been developed by Lucara, Debswana and Boteti District Council to include all villages in both mines' Areas of Influence (AoIs). The Lucara

SIMMP focuses on 19 villages in the KDM AoI and identifies impacts and mitigations including probability, scale, timeframe and severity.

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.

In March 2025 surveys were conducted for the FRD 2 and the underground water management pond (Segametsi, 2025). No sites of archeological or cultural importance were identified in these surveys, and the project area remains as graded “5” on the Botswana DNMM grading scale. Monitoring will continue again in 2028 during construction of FRD 3.

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 2025 Communication on Progress for the UNGC was published in July 2025. Lucara has submitted Communications on Progress every year since 2019.

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. An outcome of the review was the following recommendation: at a policy and system level, Lucara Botswana should add a focused element to their Social Management System (SMS) addressing human rights in the same way in which it frames its activities and performance with respect to the stated Sustainable Development Goals and ESG benchmark reporting. A field-based review of relevant human rights issues and a Human Rights Risk Assessment took place during December 2024 (Prizma, 2025). This study did not identify any salient issues with KDM or Lucara corporate facilities or activities but did identify access to water as a regional concern.

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), the 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 2027), and the 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 has been recognised by Canadian newspaper The Globe and Mail in their 2021, 2022 and 2023 benchmark studies of female leadership in corporate Canada. Lucara has 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 Prospectors & Developers Association of Canada Sustainability Award. In 2025 KDM was awarded the AfriSafe Excellence Award, African Mining Company of the Year. Lucara also received three Botswana Chamber of Mines (BCM) Safety Awards: Best in Total Recordable Injury Frequency Rate (TRIFR), Best in Classified Injury Frequency Rate (CIFR), and the Most Improved CIFR Award.

20.4 Mine Closure

Digby Wells developed the latest Rehabilitation and Mine Closure Plan (RMCP) and made updates in line with Section 65 of the *Botswana Mines and Minerals Act* (1999) and South Africa's *National Environmental Management Act*, 1998 (Act No. 107 of 1998) (Digby Wells, 2021). The mine is obliged to develop and implement a mine closure and rehabilitation plan during the life of mine and to ensure that the mining lease area is progressively rehabilitated and ultimately reclaimed at the end of life of mine to the satisfaction of the Botswana 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 provides KDM with annual reviews and closure cost estimations, the last of which was submitted in November 2024 (Digby Wells, 2024). This estimate is prepared using a combination of first principal calculations, benchmarked local and southern African rates, and rates / productivities supplied by Lucara Botswana. The general reclamation plan includes:

- Deconstructing all on-site infrastructure outside the 33 kV substation and powerline which shall be handed over to Botswana government;

- Capping of mine shafts and boreholes;
- Disposal of liners, steel, rubble, and other waste materials into on-site facilities and/or the open pit;
- Reshaping the upper open pit to a 1:3 slope, and bunding the perimeter; and
- Ripping, grading, and spreading of seeded topsoil on all disturbed ground including access roads, ensuring final dumps and dams meet a 1:3 slope.

A soil balance report prepared in 2024 indicates a shortfall of soil to meet the demand of the reclamation plan and may warrant import of soil at unscheduled cost. Investigations are underway to determine if the available sand stockpile on site can be made suitable for use as a supplement.

Lucara Botswana has provided financial guarantees totalling BWP 270.0 million for reclamation obligations, consisting of a restricted deposit of BWP 70.0 million and a BWP 200 million standby letter of credit (First National Bank of Botswana Limited, 2025). As of September 30, 2025, the estimated reclamation liability is approximately US$37.9 Million including a 10% contingency.

20.5 Permitting

Lucara maintains a register for its licences and permits for KDM and the UGP. A list of permits held or in the process of being acquired by KDM is presented below. Many permits are issued annually and renewed between December and January.

Table 20-3: KDM Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIS Approval for Karowe Underground Project	DEA/BOD/CEN/EXT/ MNE 103 II(2)	03-Jun-45	Dept. of Environmental Protection	*EIA Act*
Surface Rights	LT/SLB/B/1 IV (231)	03-Jan-46	Ngwato Land Board	*Tribal Land Act*
Mining Licence	2008/L6	03-Jan-46	Dept. of Mines	*Mines & Minerals Act*
Water Rights	B6316, B6317, B6023, B6615, B6622, B5386, B 5387, B5388, B5389, B7942, B7933, B7934, B7935, B7936, B7937, B7938, B7939, B7940, B7941, B7942	03-Jan-46	Dept. of Water Affairs	*Water Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIS Approval for Mechanical Evaporators	DEA/BOD/CEN/EXT/ MNE 103 II (08)	28-Aug-29	Dept. of Environmental Protection	*EIA Act*
Borehole Certificates	Z18429, Z18430, Z18431, Z18432, Z18433, Z18434, Z18435, Z18436, Z18437, Z18438	January-2046	Dept. of Water Affairs	*Boreholes Act*
Dumps Classification	All classified	All dumps active	Dept. of Mines	*Mines, Quarries, Works and Machinery Act*
Radiation Licence	BW0315/2025	6-Nov-27	Radiation Inspectorate	*Radiation Protection Act*
Incinerator Permit	DEP 7/1/32 II	24-Jun-27	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Waste Water Treatment Plant	WMF01/2022/11/20-WWTW/Karowe Diamond Mine	24-Dec-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Landfill	WMD/22-2024/304-10/LF/Letlhakane	31-Dec-25	Dept. of Waste Management and Pollution Control	*Waste Management Act*
Salvage yard	WMF/20-2024/20-12/Letlhakane	31-Dec-25	Department of Waste Management Pollution Control	*Waste Management Act*
Permit to purchase, acquire and Possess Explosives	F001/2022 E-PPAP0594/2024	31-Dec-25	Dept. of Mines	*Explosives Act*
Permit to carry bulk explosives	E-PCE0161/2025 Vehicle No: B868BOY	30-Dec-25	Dept. of Mines	*Explosives Act*
Explosives magazine licence	E-MLF0047/2025	31-Dec-25	DME	*Explosives Act*
Explosives magazine licence	E-MLF0046/2025	31-Dec-25	Dept. of Mines	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	E-MLF0047/2025	31-Dec-25	Dept. of Mines	*Explosives Act*
Authorization for storage of fracture Explosives (Reg 46,65 and 66)	00003647A	31-Dec-25	DME	*Explosives Act*
Permit to import and possess explosives	E-PIPE0584	31-Dec-25	DME	*Explosives Act*
Application for restricted blasting licence		N/A	Dept. of Mines	*Explosives Act*

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Permit to carry explosives in Bulk	Vehicle No: B681BMU E-PCE0163/2025	30-Dec-25	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	E-LMO0033/2025 Vehicle No: B339BPM	31-Dec-25	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	Vehicle No: B429BJB E-PCE0162/2025	31-Dec-25	Dept. of Mines	*Explosives Act*
Licence to manufacture explosives	E-LMO0034/2025 Vehicle No: B693BRO	31-Dec-25	DME	*Explosives Act*
Box storage for conveyance and Storage of explosives	F01/22 F02/22 F03/22 F04/22	As and when required	Dept. of Mines	*Explosives Act*
Blasting Licence for magazine master	In Place	valid and appointment renewed annually	Dept. of Mines	*Explosives Act*
Airstrip Licence	B509	Jun-26	Civil Aviation	*Aviation Act*
Generator Licences	CBERA 7/2/93 I	No Expiry	Botswana Energy Regulatory Authority (BERA)	*BERA Act*
Solar photovoltaic plant	CBERA 7/2/89 I	No Expiry	BERA	*BERA Act*
Standby Generator Licences	CBERA 7/2/72 I	No Expiry	BERA	*BERA Act*
Licence to possess and use radioactive sources	BW0611/2024	31-Jul-26	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 44:04
Kibble Winder 10 - 069 - Production shaft	M35 M 1	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Stage Winder 10 - 071- Ventilation shaft	M35 M 1 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Stage Winder 10 - 087/88 Ventilation shaft	M35 M I	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Vertical Shaft Mucker (VSM)	M35 M 1 (33)	14-Oct-25	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Approval letter for charging units	11-May-22	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	M35 M (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 XXVII (55)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:02
Chemical clearance letter	HSM 6/6/25 V (1)	N/A	Dept. of Public Health	*Public Health Act*
Kibble Winder 10 - 069 - Production shaft	DOM 6/13/5 I(13)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:03
Kibble Winder 10 - 0399 – Ventilation shaft	DOM 6/13/5 I(12)	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-25	Dept. of Mines	*Explosives Act*
Permit to carry explosives in Bulk	E-PCE0161/2024	31-Dec-25	Dept. of Mines	*Explosives Act*
Kibble Winder 10 - 069 Production shaft	DOM-6/13/ 51(13)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Approval of Diesel Engine Unit use in Underground Sinking Operation	DOM - 6/13/ 51 (15)	N/A	Dept. of Mines	*Mines, Minerals, Works and Machinery Act* Cap 44:04
Permit to export/import/transit controlled ozone depleting substances	00000696A	31-Dec-25	Metrological Services	*National Metrological Services Act* of 2014
Chemical clearance letter	HSM 6/6/15 V (1)	31-Dec-25	Dept. of Public Health	*Public Health Act*
Remotely Piloted Aircraft Certificate	RPA (H)950 RPA (H)958 RPA (H)957 RPA (H)951	28-Feb-26	Civil Aviation Authority of Botswana (CAAB)	*Civil Aviation Act*
Remotely Piloted Aircraft Certificate	RPA (H)953 RPA (H)968	31-Mar-26	CAAB	*Civil Aviation Act*
General Nurse Practicing Certificate	GN. 11621 GN. 4646	31-Dec-25	Nursing and Midwifery Council of Botswana	*Nurses and Midwives Act*

Source: Lucara (2025)

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 Capital Cost Summary

The capital cost estimate was prepared using a combination of historical costs, current commitments and contracts, and first principles estimates. First principles estimates apply project experience and use 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.

The capital cost estimate consists of three parts:

- Pre-production period - Incurred – UGP capital costs prior to the effective date of the FS (September 30, 2025) of US$436.0 million;
- Pre-production period – To Completion – Estimated UGP capital costs from the effective date of the FS until commercial production of US$343.2 million; and
- Sustaining – Estimated UGP capital costs occurring after commercial production (March 1, 2028), as well as capital costs for existing infrastructure constructed prior to the UGP, totalling US$240.9 million.

Total UGP specific pre-production capital costs (incurred plus to completion), are estimated to be US$779.2 million. These costs do not include current and future sustaining costs for the existing OP operations.

Remaining pre-production contingency for the UGP is US$52.4 million, or 15% of remaining spend. Closure costs amount to US$29.9 million, are included in sustaining capital, and were assumed to occur concurrent to plant closure.

The cost estimate is an AACE Class 3 Estimate.

Costs are expressed in 2025 US$ with no escalation unless stated otherwise. Foreign exchange rates of BWP14.00:US$1.00 and ZAR17.50:US$1.00 are used where applicable.

These costs are summarised in Table 21-1.

Table 21-1: Capital Cost Summary

WBS	Capital Costs	Pre-Production			Sustaining (US$M)	LOM Total (US$M)	Weight (%)
		Incurred (US$M)	To Completion (US$M)	Subtotal (US$M)			
1000	Mining	254.6	183.9	438.5	59.4	497.9	53
2000	Site Development	17.6	3.8	21.4	6.8	28.2	3
3000	Process Plant	-	-	-	-	-	0
4000	Tailings and Mine Waste	-	-	-	42.4	42.4	5
5000	On-site Infrastructure	11.6	0.2	11.8	92.2	104.0	11
6000	Buildings and Facilities	2.5	0.7	3.3	-	3.3	0
7000	Off-site Infrastructure	27.0	-	27.0	-	27.0	3
8000	Project Indirects	15.9	10.6	26.5	2.6	29.1	3
9000	Owner Costs	106.8	91.6	198.4	0.8	199.2	21
Subtotal		**436.0**	**290.8**	**726.8**	**204.1**	**930.9**	**100**
10000	Contingency	-	52.4	52.4	7.0	59.3	
11000	Closure	-	-	-	29.9	29.9	
Total Capital Costs		**436.0**	**343.2**	**779.2**	**240.9**	**1020.1**	

Notes:
*Numbers may not add due to rounding.

21.1.2 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 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) and 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 Section 24.1.2 of this Report;
- UG mine development and construction 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 contractors.

The following key parameters apply to the capital estimate:

- Estimate Class: The capital cost estimate is considered an AACE Class 3 Estimate;
- Estimate Effective Date: The effective date of the capital estimate is September 30 2025. 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 USD =	14.0	BWP
	17.5	ZAR

21.1.2.1 Quantity Development

The capital estimate has been developed largely from engineering quantities obtained from engineering drawings, lists, Material Take Offs (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 (%)
Mining	m/m^3	0
Civil and Earthworks	m/m^3	20
Concrete	m^3	10
Structural Steel	t	5
Architectural	m^2	5

Description (Discipline)	Unit	Growth Allowance (%)
Mechanical	ea	N/A
Platework	t	10
Piping	m	10
Electrical Bulks	m	10
Instrumentation Bulks	m	10
Ventilation Bulks	ea/m	10

21.1.2.2 Equipment

Equipment costs were developed from actual costs, Purchase Orders, budgetary quotes and database costs from comparable projects. In general, 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 for fixed equipment; and
  - 3% of the sum of the base price and project specific accessories for UG mobile equipment, which requires additional support for assembly after slinging underground.
- First fills, identified as either a quoted cost or 0.5% of the sum of the base price and project specific accessories.

21.1.2.3 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 the QP based on experience.

21.1.2.4 Installation

Direct field installation manhours for permanent infrastructure installations have been estimated 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 on-site historical performance or past 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.

21.1.2.5 Labour

UG mining contractor staffing levels are built up on a combination of existing contracts for shaft sinking and lateral development, and estimated manhours required for construction and production activities occurring within the project schedule. As such, mining manpower fluctuates throughout the capital development period. A summary of the labour requirements is located in Figure 16-96.

Where existing contracts are not already in place as the basis of the labour cost estimate, the average labour rates of the existing contracts on site for shaft sinking and lateral development contracts are used. The following key considerations have been made for the contract labour rate estimate:

- Work is performed on a 2 weeks on: 1 week off roster for local labour and a 6 weeks on: 3 weeks off roster for expat labour. Scheduled overtime has been considered in the base labour rates;
- Expats consist of approximately 35% of the labour force, with local labour accounting for the remaining 65%;
- Travel costs have been included in the labour rate; and
- Indirect staffing and overheads are included as a factor on the direct costs.

The Owners Team staffing plan is based on labour histograms developed by the Owner's operational readiness team. It provides labour details on the phased progression from a Project EPCM team to an Operations team responsible for the shafts, fixed infrastructure and drawpoint operations. Varying work schedules are utilised based on department and responsibilities. Typically, this consists of one of the following:

- 5 days on, 2 days off – Management, Technical Services, Admin;
- 4 days on, 4 nights on, 4 days off – Mining Operations; or

- 6 weeks on, 3 weeks off – Expats.

Lucara uses a tiered salary system, where a base salary consists of:

- Base salary; and
- Scheduled overtime due to the work schedule.

Local labour salaries also include provision for the following burdens:

- Leave allowances;
- Housing and car allowance;
- Medical and insurance; and
- Gratuity.

In general, expat labour salaries include provision for the following burdens:

- Income taxes; and
- Insurance.

Burdens vary by salary tier and account for approximately 60% of the base salary. All Owner's Team costs as based on fully burdened rates.

21.1.3 Mining Capital Costs

Mining capital costs consist of primarily UG mining activities; there are no capital costs associated with OP operations. Incurred 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 mining capital costs contain a mix of actual costs, purchase orders, budgetary quotes from local vendors, and database costs from comparable projects. Time-related costs for development or infrastructure installations include third party vendor and contractor estimates.

UG capital costs are summarized in Table 21-4.

Table 21-4: UG Mining Capital Costs

WBS	UG Mining Capital Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1100	Surface Infrastructure	59.4	6.2	-	65.6	13
1200	Shaft Sinking and Infrastructure	182.9	37.1	-	219.9	44
1300	UG Development	-	60.8	16.2	76.9	15
1400	UG Equipment	5.9	36.0	35.3	77.2	16
1500	UG Infrastructure	6.4	29.2	7.9	43.5	9
1600	Capitalized UG Operating Costs	-	14.6	-	14.6	3
1000	**Total Mining**	**254.6**	**183.9**	**59.4**	**497.9**	**100**

21.1.3.1 Surface Infrastructure

Surface infrastructure capital costs are summarised in Table 21-5.

Table 21-5: Mine Capital - Surface Infrastructure

WBS	Surface Infrastructure	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1110	Hoist House	30.8	1.4	-	32.2	49
1120	Production Shaft Head Frame	6.0	-	-	6.0	9
1130	Services	1.0	0.2	-	1.3	2
1140	Ventilation Shaft Head Frame	2.7	-	-	2.7	4
1150	Electrical Supply and Distribution	7.7	0.1	-	7.8	12
1160	Ventilation and Cooling	8.0	3.5	-	11.5	18
1170	Compressed Air	1.0	-	-	1.1	2
1180	Buildings and Facilities	0.2	0.8	-	1.0	1
1190	Construction Support Facilities	2.0	0.1	-	2.1	3
1100	**Total**	**59.4**	**6.2**	**-**	**65.6**	**100**

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. Major projects required for the final stages of construction and start of operations such as the man and materials winder and bulk air cooler are substantially complete, with only the main fan installation remaining to be installed in 2026.

The majority of these costs to completion have been defined through contracts or purchase orders specific to the remaining tasks.

21.1.3.2 Shaft Sinking and Infrastructure

Shaft Sinking and Infrastructure capital costs are summarised in Table 21-6.

Table 21-6: Mine Capital – Shaft Sinking and Infrastructure

WBS	Shaft Sinking	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1210	Common Preliminaries and Generals	-	-	-	-	0
1220	Shaft Pre-Sink	33.3	0.4	-	33.7	15
1240	Shaft Main Sink	119.8	27.1	-	146.9	67
1260	Shaft Equip and Commission	8.1	9.6	-	17.7	8
1280	Shaft Operating Indirects	21.6	0.1	-	21.7	10
1200	**Total**	**182.9**	**37.1**	**-**	**219.9**	**100**

The pre-sink of both the P/S and V/S are complete, while main sink activities have achieved target elevations. The costs to complete are primarily associated with equipping and commissioning the production shaft, while lateral development and construction activities are performed from access via the ventilation shaft. Costs to complete are derived from an existing contract price prepared by the sinking contractor, updated to account for approved change orders.

21.1.3.3 UG Development

UG development capital costs are summarised in Table 21-7.

Table 21-7: Mine Capital - UG Development

WBS	UG Development	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1310	Station 1 Drill Horizons	-	0.8	1.7	2.5	3
1320	Station 2 Drill Horizons	-	4.8	0.1	4.9	6
1330	Station 3 Fine Ore Storage	-	2.8	-	2.8	4
1340	Station 4 Extraction Drives	-	15.4	0.7	16.1	21
1350	Station 5 Shaft Load Out	-	3.9	1.6	5.5	7
1360	Station 6 Shaft Bottom	-	1.1	1.6	2.7	3
1370	Vertical Development	-	5.2	-	5.3	7
1380	Development Contractor General	-	19.9	9.5	29.4	38
1390	Construction	-	6.8	1.0	7.8	10
1300	**Total**	**-**	**60.8**	**16.2**	**76.9**	**100**

UG development includes all work completed by the contractor inclusive of lateral development, vertical development, and construction support tasks. These construction tasks primarily involve the establishment of sumps, substations and ventilation controls, as well as the establishing piping and electrical distribution networks. Major construction tasks are excluded and carried separately. UG development does not include shaft sinking which is covered in Section 21.1.3.2.

Costs for underground development are based on the selected UG development contractor rates. Development costs account for the labour, equipment, materials, and supervision required to drive all lateral and vertical development prior to commercial production. Mobile equipment maintenance and operation is performed by the contractor; mobile equipment in general is supplied by the Owner as outlined in Section 21.1.3.4. Free-issued materials such as shotcrete and ground support are supplied by the Owner as outlined in Section 21.1.10.

Lateral and vertical development unit costs are summarised below:

- Lateral development blended unit cost (excluding equipment ownership, fuel, explosives, and power) - US$4,500/m:
  - Heading sizes range from 5.0 m W x 5.0 m H to 9.0 m W x 5.5 m H.
- Raise bore development blended unit cost - US$3,000/m:
  - Raise size range from 2.1 m to 4.0 m in diameter.
- Drop raising blended unit cost – 1,500/m:
  - Drop raises range in size from 2.4 x 2.4 m to 4.0 x 4.0 m.

21.1.3.4 UG Mobile Equipment

UG mobile equipment capital costs are summarised in Table 21-8 and exclude shaft sinking equipment.

Table 21-8: Mine Capital – UG Equipment

WBS	UG Equipment	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1410	Drilling	1.9	19.4	3.4	24.7	32
1420	Charging	-	2.0	-	2.0	3
1430	Loading	1.9	5.8	3.9	11.6	15
1440	Hauling	-	-	-	-	0
1450	Ground Support	-	1.9	1.2	3.0	4
1460	Services	1.4	3.0	-	4.3	6
1470	Ancillary	0.8	4.0	-	4.7	6
1480	Equipment Overhauls and Replacements	-	-	26.9	26.9	35
1400	**Total**	**5.9**	**36.0**	**35.3**	**77.2**	**100**

UG mobile equipment quantities and costs were determined through a build-up of contractual requirements and estimated quantities based on productivities. The costs for the majority of UG mobile equipment are based on existing contracts and project specific quotes.

The Owner will supply the lateral development and production drilling contractors with the UG mobile equipment utilised for their scope. This approach reduces costs and ensures equipment is owned by the Owner at the conclusion of the contractor's scope, in order to perform drawpoint operations and mine maintenance activities. The UG mobile equipment required by the UG construction contractor is assumed to be supplied by the contractor and has been excluded from this estimate.

The equipment fleet will be purchased through a mixture of outright purchases and financing agreements, depending on the supplier and order timing. In general, equipment is assumed to be purchased with a 20% down payment, 12 months in advance of requirement on site, with the remaining 80% paid upon delivery. Alternatively, select equipment is assumed to be financed at a rate of 8%, with a 15% deposit. Equipment will be brought on site three months ahead of UG requirements. Mobile equipment purchases are assumed new, although some select used and rebuilt gear has been sourced for the project

All mobile equipment will come supplied with the necessary safety features required by local and mine site regulations including fire protection, door interlocks, and emergency steering. Where possible, equipment will be outfitted with enclosed cabins and air conditioning to protect against heat stress and air quality concerns. Harsh water packages, auto-lubrication and foam filled tires

will be applied where possible to reduce downtime. The production LHDs will be equipped with tele remote capabilities to allow operators to sit in a control room away from the drawpoint hazards. The jumbos primarily performing development drilling activities will be outfitted with operator assist packages.

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.

Table 21-9 lists the LOM equipment purchases, rebuilds, and replacements, as well as fleet purchased to date.

Table 21-9: Mine Equipment Capital Costs

Equip Type	Manufacturer / Model	On Site	Total Required	Outright Purchase	Financed [1]	Unit Cost (US$M) [2]	Rebuilds	Replacements
Drilling and Charging								
Jumbo - 2 Boom - Small	Sandvik - DD321	2	5	1	4	1.4	0	0
Jumbo - 2 Boom	Sandvik - DD422i	0	2	2	0	1.7	1	1
Cable Bolter	Sandvik - DS422i	0	1	0	1	1.6	0	0
Longhole Drill - ITH (Dev)	Sandvik - DU311	0	2	2	0	1.6	0	0
Longhole Drill - ITH (Prod)	Sandvik - DU311 - TK	0	4	4	0	1.1	0	0
Small Explosives Truck	Fermel - Liberator Boom Basket Charger - Mk iii	0	2	2	0	0.4	0	0
Large Explosives Truck	Fermel - Liberator Evo Iii Knuckle Boom Basket Charger	0	2	2	0	0.5	0	0
Load and Haul								
LHD (7 t / 2.8 m^3)	CAT R1300	1	1	1	0	0.8	2	1
LHD (12.5 t / 5.5 m^3)	CAT - R1700G	1	1	1	0	0.6	0	1
LHD (17 t / 8.4 m^3)	Sandvik - LH517i	0	4	2	2	1.2	6	5
LHD (21 t / 9.5 m^3)	Sandvik - LH621	0	2	0	2	1.4	3	3
Ground Support, Services, Ancillary Equipment								
Telehandler UG	Magni TH5.8	2	4	4	0	0.3	2	2
UG Integrated Tool Carrier	Volvo L90F	0	3	3	0	0.5	2	2
Boom Truck	Fermel - Liberator RORO	0	3	3	0	0.4	1	1
Transmixer	Fermel - Liberator 4 cu Concrete Mixer	0	2	2	0	0.4	0	0
Shotcrete Sprayer	Fermel - Liberator Evo Iii Robotic Spraymech	0	1	1	0	0.8	1	1
Shotcrete Cassettes	Fermel - Shotcreter Cassette	0	1	1	0	0.1	0	0
Secondary Breakage Drill	Sandvik - DB331	0	1	1	0	1.2	1	0

Equip Type	Manufacturer / Model	On Site	Total Required	Outright Purchase	Financed [1]	Unit Cost (US$M) [2]	Rebuilds	Replacements
Rock Breaker Attachment	Xrock XB36 with XC80HD Carriage	0	2	2	0	0.2	3	3
Fuel/Lube Truck	Fermel - Liberator RORO	0	1	1	0	0.4	0	0
Service Truck	Fermel - Liberator RORO	0	2	2	0	0.4	0	0
Utility Vehicle	Mahindra S6 Mining 4x4 Double Cab	0	17	17	0	0.0	0	0
Surface Equipment [3]								
Surface Telehandler 24T	Magni HTH 24.11	2	2	2	0	0.4	0	0
Surface Telehandler 10T	Magni HTH 10.10	1	1	1	0	0.3	0	0
Surface Warehouse Forklift	Toyota 8FD30	1	1	1	0	0.1	0	0
Concrete Mixer Truck	FAW – J5N 33,340 6cu Mixer	3	3	3	0	0.1	0	0
Front end loader	SDLG L958F	1	1	1	0	0.5	1	0
Tractor	New Holland - TD 5.75/4 Cab 55 kW	1	1	1	0	0.0	0	0
Combi - 16s	Toyota -Hiace	2	2	2	0	0.1	5	5
Utility Vehicle - Surface	Various	7	7	7	0	0.0	7	7
Hiab Loader Crane	HIAB - X-CL19B-2	1	1	1	0	0.1	1	1

Notes:

1. Equipment financing of a portion of the fleet has been incorporated into the cost estimate; financing costs vary by agreement, but in general, financing defers pre-production capital costs and increases sustaining capital costs;
2. Unit Cost represents a typical unit's total capital cost inclusive of capital spares, freight, assembly, etc. when purchased outright. This cost can vary within each equipment type due to the timing of procurement or variations in equipment specifications; and
3. Surface equipment costs are associated with equipment supporting solely the UGP. Incurred costs have typically fallen under surface infrastructure and shaft sinking cost categories as procurement was performed through the shaft sinking contractor. Future equipment replacements and rebuilds of the surface equipment supporting the UGP are included in WBS 1480 – Equipment Overhauls and Replacements. Equipment capital costs associated with the various other surface equipment supporting the Karowe site are included in site budgets under On-site Infrastructure - WBS 5000.

21.1.3.5 UG Infrastructure

UG infrastructure capital costs are summarised in Table 21-10.

Table 21-10: Mine Capital - UG Infrastructure

WBS	UG Infrastructure	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1510	Electrical Distribution	1.8	6.2	1.1	9.1	21
1520	Material Handling	2.7	11.2	1.1	14.9	34
1530	Maintenance Shop and Services	-	1.7	-	1.8	4
1540	Miscellaneous Infrastructure	-	-	-	-	0
1550	Mine Dewatering	1.4	2.3	5.5	9.2	21
1560	Mine Ventilation	0.4	1.1	-	1.6	4
1570	Piping	-	0.6	-	0.6	1
1580	Instrumentation and Communication	-	0.9	-	0.9	2
1590	Safety	-	5.2	0.2	5.4	13
1500	**Total**	**6.4**	**29.2**	**7.9**	**43.5**	**100**

All major infrastructure costs were developed from a mix of actual costs, purchase orders, budgetary quotes from local vendors where possible, and database costs. Incurred 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 electrical cables.

The materials handling costs include the UG crusher and conveyor system and skip loading infrastructure. These costs are inclusive of concrete, structural steel, platework, mechanical equipment and low voltage electrical equipment and distribution supply and installation. Costs associated with power distribution supply and installation, 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, local fire suppression, and tooling. These costs also include nearby auxiliary infrastructure such as lunchrooms and ablutions.

The mine dewatering cost include the supply and installation of all underground dewatering systems beyond the shafts including concrete, piping, pumps, structural steel, low voltage power distribution and instrumentation. The system consists of a network of sumps, as well as three pumping stations and a flood drift system.

Mine ventilation includes the supply of all permanent fans, ducting, instrumentation and controls, man doors, fire doors, air locks, fan bulkheads, and regulators. The costs for installing auxiliary fans and ventilation controls are carried under the lateral development capital costs. Costs associated with electrical installations and chamber excavation are carried elsewhere.

Piping estimates are limited to primarily general piping requirements within infrastructure as well as fire water pressure reducing stations. Piping supply and installation costs for the distribution of compressed air, service water, potable water, fire water and dewatering services are included in the scope of the lateral development contract.

Network cabinets, PLCs and other instrumentation typically installed near UG substations have been included in the instrumentation and communication category. The UG communications systems costs account for leaky feeder and fiber distribution throughout the mine, as well as a private LTE system to support teleremote operations at the 310 L.

The UG safety system estimates include costs for mine refuge stations and fire suppression equipment such as monitoring panels, detection systems, and controls. Geotechnical monitoring equipment is also included in this category which consists of drilling instrumentation holes, instrumentation supply and installation, as well as various monitoring programs.

21.1.3.6 Capitalized UG Operating Costs

Capitalized operating costs refer primarily to the drilling and blasting contractor costs prior to commercial production. The costs associated with loading and hauling ore from the drawpoints prior to commercial production are included separately under Owner's Cost build-ups.

Table 21-11: Mine Capital – Capitalized Operating Costs

WBS	Capitalized Operating Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
1610	Production Stoping	-	14.6	-	14.6	100
1600	**Total**	**-**	**14.6**	**-**	**14.6**	**100**

21.1.4 Site Development

Earthworks and civils costs associated with the UGP pad are complete, inclusive of geotechnical programs. The surface dewatering pond and initial phase of mechanical evaporators is substantially complete, with remaining costs scheduled to occur primarily in the sustaining capital period. A hydrological kimberlite depressurization drilling program is planned for UG prior to developing into the orebody.

Table 21-12: Site Development Costs

WBS	Site Development	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
2100	Bulk Earthworks	11.8	-	-	11.8	42
2200	Site Roads	-	-	-	-	0
2300	Surface Water Management	3.2	0.6	6.8	10.6	38
2400	Dewatering	-	3.2	-	3.2	11
2500	Core Hole Drilling	2.5	-	-	2.5	9
2000	**Total Site Development**	**17.6**	**3.8**	**6.8**	**28.2**	**100**

21.1.5 Tailings

Tailings facility costs are associated with the establishment of new facilities or major expansions of the existing facilities on site. Costs associated with routine projects on the FRD are capitalized and covered under the existing on-site infrastructure budget.

Sustaining costs for tailings consist of an additional lift to FRD 2 where tailings deposition is currently underway, as well as an expansion of FRD 3. Design details are discussed in Section 18.4.5.

All tailings facility costs are considered sustaining capital, as costs scheduled prior to completion of the UGP in Q1 2028 are primarily associated with ongoing open pit operations and the processing of existing stockpiles.

Budgetary capital estimates for the FRD expansion have been generated by KDM and verified by the QP. While both the CRD and the WRSF will be expanded as part of the UGP, these activities are minor and captured under operating budgets.

Table 21-13: Tailings Facility Costs

WBS	Tailings Facility	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
4100	FRD - Fines	-	-	42.4	42.4	100
4200	CRD - Coarse	-	-	-	-	0
4300	Waste Rock Storage Facility	-	-	-	-	0
4000	**Total Tailings**	**-**	**-**	**42.4**	**42.4**	**100**

21.1.6 On-site Infrastructure

All on-site infrastructure such as electrical power and service water supply and distribution projects to support the UGP are substantially complete. Sustaining capital costs for on-site infrastructure are associated with the annual budgets for maintaining all existing infrastructure constructed prior to the UGP. This includes facilities such as the process plant, tailings facility, site security system, site services, and access road.

All existing on-site infrastructure costs are considered sustaining capital, as costs scheduled prior to completion of the UGP in Q1 2028 are primarily associated with ongoing open pit operations and the processing of existing stockpiles.

Table 21-14: On-site Infrastructure Costs

WBS	On-site Infrastructure	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
5000	Existing On-site Infrastructure	-	-	92.2	92.2	89
5100	Electrical Supply and Distribution	8.8	0.2	-	9.0	9
5200	Water Supply, Distribution, and Treatment	2.3	-	-	2.4	2
5300	Waste Collection and Treatment	0.5	-	-	0.5	0
5400	IT and Communications	-	-	-	-	0
5000	**Total On-site Infrastructure**	**11.6**	**0.2**	**92.2**	**104.0**	**100**

21.1.7 Buildings and Facilities

Buildings and facilities on site are established with remaining costs associated with additional mine rescue equipment requirements as underground development and construction activities progress.

Table 21-15: Buildings and Facilities Costs

WBS	Building and Facilities	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
6100	Training Center	-	-	-	-	0
6200	Workshop and Warehouse	0.2	-	-	0.2	5
6300	Mine Rescue Centre	1.7	0.7	-	2.4	73
6400	Offices	0.5	-	-	0.5	15
6500	Change house	0.2	-	-	0.2	6
6600	Access, Fencing, and Traffic Management	-	-	-	-	0
6000	**Total Buildings and Facilities**	**2.5**	**0.7**	**-**	**3.3**	**100**

21.1.8 Off-site Development

Off-site development costs for the power transmission line and off-site accommodation facilities are summarised in Table 21-16.

Table 21-16: Off-Site Development Costs

WBS	Off-Site Development	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
7100	Power Transmission Line	22.3	-	-	22.3	83
7200	Off-site Accommodations	4.7	-	-	4.7	17
7000	**Total Off-site Development**	**27.0**	**-**	**-**	**27.0**	**100**

21.1.9 Project Indirects

Project indirects cover camp catering, office rentals, bussing, material haulage, civil material testing, freight and freight forwarding services, and charter flights for personnel. After commercial production in Q1 2028, project indirects associated with UG contractors completing remaining scope has been capitalized, while project indirects for the Owner's Team are considered operating costs.

Table 21-17: Project Indirects Costs

WBS	Project Indirects	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
8100	On-site Contract Services	8.0	8.2	2.6	18.8	65
8200	Temporary Facilities and Utilities	-	-	-	-	0
8300	Contractor Indirects	0.3	0.6	-	0.9	3
8400	Freight	6.8	0.5	-	7.4	25
8500	Temporary Accommodations and Expenses	0.8	1.3	-	2.0	7
8000	**Total Project Indirects**	**15.9**	**10.6**	**2.6**	**29.1**	**100**

21.1.10 Owner's Cost

Owner's costs are classified as the management, oversight and site operation costs that are instrumental to develop and operate the UGP. These costs are capitalized during the construction phase. In general, Owner's costs that continue beyond the project phase are then incorporated into the mining operating costs. Only specific free-issued materials such as ground support, concrete and shotcrete associated with contractor development and construction can be capitalized in the sustaining cost period.

Owner's costs include:

- Owner's labour;
- 3rd party engineering and consulting services;
- Free-issue materials including fuel, power, explosives, ground support, concrete, and shotcrete;
- Project taxes and insurance;

- Safety and Training;
- Site Services;
- Equipment fleet maintenance; and
- Fixed equipment maintenance (shafts, BAC, crusher and conveyor, etc.) and major component replacements.

The Project's external EPCM team transitioned to roles within Lucara at the end of Q1 2025. The EPCM team is fully integrated within the Owner's team and labour structure, with all procurement and construction management costs reporting under the pre-production G&A cost category until project completion. Remaining costs to completion under the EPCM category are primarily associated with external support, studies, and detailed engineering.

In general, cost estimates for the Owner's capital costs have been generated by the Owner's operational readiness team and verified by the QP.

Table 21-18: Owner's Costs

WBS	Owners Capital Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
9100	Pre-Production General and Administration	14.6	41.2	-	55.8	28
9200	Operational Charges	15.9	16.0	-	32.0	16
9300	Engineering, Procurement, and Construction Management	56.7	4.7	-	61.4	31
9400	Equipment Supply and Maintain	0.9	2.0	-	2.9	1
9500	Free-Issue Materials	18.6	27.7	0.8	47.2	24
9000	**Total Owners Costs**	**106.8**	**91.6**	**0.8**	**199.2**	**100**

21.1.11 Contingency

Project Contingency costs consist of three categories:

- Owners – General project contingency of approximately 10% of the estimated cost to completion. These costs cannot be identified or itemized at this stage of the Project, but they are expected to be incurred;
- Project Change Log – A Project Change Log has been developed which outlines specific potential changes to existing project scope or contracts. These costs are not planned and

not contractual but are expected to be incurred based on historical performance on the project. These costs include, but are not limited to, additional labour requirements to achieve key project milestones, elevated labour rates to source key highly-skilled operators to maintain project schedules, or the costs associated with contractual claims. The Project Change Log also includes approximately $2 million of salvage value associated with the expected sale of select shaft sinking equipment; and

- Sustaining – General project contingency of approximately 10% of the estimated sustaining capital costs at the underground working area. This excludes sustaining capital costs on the existing on-site infrastructure and tailings facilities, as these costs are more defined and based on historical budgets.

No provision is made, or contingency allowed for external project risk, unidentified changes to the scope or work, general project escalation, or foreign exchange fluctuations. Limited contingency allowance for internal project risk has been allowed as defined in the Project Change Log. Remaining pre-production contingency is approximately 15% of the remaining cost to completion.

Table 21-19: Contingency

WBS	Contingency Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
10110	Owners	-	30.0	-	30.0	50
10130	Project Change Log	-	22.4	-	22.4	38
10140	Sustaining Capital Contingency	-	-	7.0	7.0	12
10000	**Total**	**-**	**52.4**	**7.0**	**59.3**	**100**

21.1.12 Closure Cost Estimate

Closure costs were originally prepared by Digby Wells in 2019 in preparation for the 2019 Feasibility Study and have been most recently updated in November 2024 for the KDM with revised rates and quantities.

Closure costs have been estimated using a unit rate approach against the planned KDM site infrastructure, inclusive of the cost of demolishing, removing and rehabilitating each component of the site infrastructure.

Lucara Botswana has provided financial guarantees totalling BWP 270.0 million for reclamation obligations, consisting of a restricted deposit of BWP 70.0 million and a BWP 200 million standby letter of credit (First National Bank of Botswana Limited, 2025). The closure costs have been estimated at US$29.9 million, which is an estimated total cost of US$34.9 million less the restricted deposit of US$5 million (BWP 70 M), excluding contingencies.

These closure costs encompass the entire KDM site inclusive of the open pit, UGP, tailings facilities, process plant and all related infrastructure.

Table 21-20: Closure Costs

WBS	Closure Costs	Pre-Production		Sustaining (US$M)	LOM Total (US$M)	Weighting (%)
		Incurred (US$M)	To Completion (US$M)			
11110	Pit Buildings and Surface	-	-	2.8	2.8	9
11120	Open Pit and Dumps	-	-	11.4	11.4	38
11130	Slimes and Dams	-	-	10.3	10.3	34
11140	UG	-	-	1.4	1.4	5
11150	Monitoring	-	-	2.1	2.1	7
11160	Project Management	-	-	1.9	1.9	6
11000	**Total Closure Costs**	**-**	**-**	**29.9**	**29.9**	**100**

21.2 Operating Cost Estimate

21.2.1 Operating Cost Summary

As the KDM is currently in operation, the operating cost estimates for OP mining, processing and other power, G&A, cost of sales, and corporate charges, were prepared using forecast operating budgets provided by Lucara as verified by the QP. 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 on-site mining, processing, coarse and fine residue deposition, and administration costs. Off-site, in-country corporate costs such as Lucara Botswana management and cost of sales have been provided by Lucara and are included as sales and corporate costs. Out of country corporate costs associated with Lucara Diamond have been excluded from the operating costs.

All UGP costs incurred during the construction phase are capitalized and form part of the capital cost estimate. All other KDM operating costs incurred during the construction phase to support the current operations are included as part of operating costs.

Operating costs are presented in 2025 US dollars (US$ or $), with no consideration for escalation or inflation. Total operating costs over the LOM are US$1,315 million and are summarised in Table 21-21.

Table 21-21: Breakdown of Estimated Operating Costs

Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Open Pit Mining Costs [3]	10.2	20.3	1.1	18.0	2
UG Mining Costs	42.7	427.3	28.5	15.0	32
Stockpile Rehandle Costs	3.8	19.3	7.6	2.5	1
Process Costs	24.7	345.3	37.3	9.3	26
Other Power Costs	7.1	99.2	37.3	2.7	8
G&A	18.5	259.1	37.3	6.9	20
Cost of Sales	3.6	49.8	37.3	1.3	4
Corporate Charges (Botswana)	6.8	95.0	37.3	2.5	7
Total	**94.0**	**1,315.4**	**37.3**	**35.3**	**100**

Notes:
1. Average cost per year in which costs occur;
2. Tonnes processed in relation to operating cost category; and
3. Includes rehandle costs incurred during open pit operations and demobilization costs as the end of open pit mine life.

Main operating costs component assumptions are shown in Table 21-22.

Table 21-22: Main Cost Assumptions

Item	Units	Base	Source
Exchange Rates, Escalation, and Taxes			
South African Rand	xZAR:1USD	17.50	Historical Average
Botswana Pula	xBWP:1USD	14.00	KDM
Escalation Rate	%	0	Excluded
Value Added Tax (VAT)	%	14	BURS
Power			
Fixed Charge	BWP/month	115.05	BPC Line Power Delivered to site, excluding VAT
Demand Rate	BWP/kW	312.44	
Energy Charge	BWP/kWh	1.07	
Fuel			
Diesel Fuel, 50 ppm	BWP/L	15.11	September 15th, 2025 BERA update. Petrohyper Delivered to site, excluding VAT

Item	Units	Base	Source
Labour			
A2	BWP/month	15,409	2025 budgets, mid-range, fully burdened. 5x2 shift base case.
B1	BWP/month	18,988	
B2	BWP/month	21,282	
B3	BWP/month	25,246	
B4	BWP/month	29,664	
C1	BWP/month	41,783	
C2	BWP/month	51,375	
C3	BWP/month	64,862	
C4	BWP/month	71,619	
D1	BWP/month	95,782	
D2	BWP/month	114,634	
D3	BWP/month	140,469	
D4	BWP/month	160,950	

Operational labour rates have been estimated by applying legal and discretionary burdens to the KDM base labour rates. Lucara uses a tiered salary system with defined wage scales applied to the various operational positions based on the skill level and expected salary for existing site roles, with a provision for salaries at typical expat rates in certain key positions. Lucara's operational readiness team provided the estimate of the labour build up and costs, with verification performed by the QP. Labour rates were benchmarked against existing positions, similar positions in regional mines, as well as existing on-site contractor rates.

21.2.2 Mine Operating Cost Estimate

21.2.2.1 OP Operating Costs

OP operating costs are based on the forecast and annual budget prepared by Lucara in October 2025 and verified by the QP. OP operations are currently performed by a mining contractor. The existing contract mining rates and the annual budget were used to forecast OP operating costs in 2026. Incremental costs for mining at depth and haulage to WRSF destinations were applied according to the existing contract. The combined OP and UG mining schedule were used to determine the stockpile rehandling requirements, with the existing contract rates being used to determine the costs. The OP mining operating costs are listed in Table 21-23.

Table 21-23: 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
	US$M	US$M	Mt	US$/t	%
Open Pit Mining	10.2	20.3	1.1	18.0	51
Stockpile Rehandle	3.8	19.3	7.6	2.5	49
Total	**6.6**	**39.6**	**8.8**	**4.5**	**100**

21.2.2.2 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 activities:

- Lateral Development - costs are associated with standard free-issued materials (explosives, fuel, power) provided to the lateral development contractor, as well as free-issued materials for rehabilitation. Lateral development contract costs as well as free-issue ground support and shotcrete are capitalized and not included in UG operating costs;
- Drill and Blast – costs related to a contractor performing longhole drilling and blasting in the area of blast assisted caving operations. The contractor will perform maintenance on their own equipment and provide their own task specific materials, with the Owner's Team providing standard free-issued materials (explosives, fuel, power);
- Load and Haul – costs associated with mucking ore from the drawpoints to the rockbreaker station, including secondary blasting and mobile breaking. All activities will be performed by the Owner;
- Crushing and Hoisting – costs related to the operation of the UG rockbreaking station, crusher, conveyors, and shafts. All activities will be performed by the Owner;
- Surface Haulage – costs related to the contractor performed transfer for material from stockpiles on surface at the UGP pad to the ROM facility at the process plant. The contractor will perform maintenance on their own equipment and provide their own task specific materials as part of the contract rate. Surface haulage costs include fuel costs estimated from existing contractual rates and historical requirements;
- Mine Maintenance – costs related to the maintenance of shafts, UG fixed equipment, and owner operated mobile equipment; and
- Mine General – costs related to mine support activities such as supervision, technical services, admin, training, safety, IT, and site services.

No contingency has been applied to UG mining operating costs.

Table 21-24: UG Mining Operating Cost Summary by Activity

Operating Costs	Average Annual [1]	Life of Mine	Tonnes Processed [2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Lateral Development [3]	0.3	2.6	28.5	0.09	1
Drill & Blast	11.7	23.4	28.5	0.82	5
Load and Haul	2.4	23.7	28.5	0.83	6
Crushing and Hoisting	6.2	62.3	28.5	2.18	15
Surface Haulage	6.0	60.3	28.5	2.11	14
Mine Maintenance	10.6	106.0	28.5	3.72	25
Mine General	14.9	149.1	28.5	5.23	35
Total	**42.7**	**427.3**	**28.5**	**14.98**	**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;
2. Tonnes processed in relation to underground operating cost; and
3. Lateral Development costs are primarily performed by a contractor and capitalized. Costs in this category are the standard free-issued materials (explosives, fuel, power) provided by the Owner's Team for both lateral development and rehabilitation activities occurring throughout the mine life.

Table 21-25: UG Mining Operating Cost Summary by Cost Category

Operating Costs	Average Annual(1)	Life of Mine	Tonnes Processed(2)	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Contractor Cost					
Surface Haulage	5.3	52.6	28.5	1.84	12
Drill & Blast	7.5	15.0	28.5	0.53	4
Owner's Cost					
Labour	15.9	158.6	28.5	5.56	37
Equipment	6.1	60.6	28.5	2.12	14
Materials	1.0	10.4	28.5	0.37	2
Explosives	4.6	9.3	28.5	0.33	2
Fuel	2.1	20.7	28.5	0.73	5
Power	5.9	59.1	28.5	2.07	14

Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Other	4.1	41.1	28.5	1.44	10
Total	**42.7**	**427.3**	**28.5**	**14.98**	**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 in relation to underground operating cost.

21.2.2.2.1 Owner's UG Mining Labour

The Owner's Team staffing plan is based on labour histograms developed by the Owner's operational readiness team. The staffing plan provides labour details on the phased progression from a Project EPCM team to a mining operations team responsible for the operation, maintenance and overall supervision and administration, of the drawpoints, materials handling system, and shafts.

Varying work schedules are utilised based on department and responsibilities. Where appropriate, labour rates include all overtime and burdens associated with shift work. Burdens vary by salary tier and account for approximately 60% of the base salary, and account for items such as gratuity, medical, leave, group insurance, and for those eligible, housing and car allowance. Expatriate labour rates benchmarked against existing contracts on site for shaft sinking and lateral development, have been used for key positions on site.

Table 21-26: UG Owner's Labour Cost Summary

Owner's Labour OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Lateral Development	-	-	-	0
Drill & Blast	-	-	-	0
Load and Haul	1.7	16.6	0.58	10
Crushing and Hoisting	4.4	44.0	1.54	28
Surface Haulage	-	-	-	0
Mine Maintenance	4.2	41.6	1.46	26
Mine General	5.6	56.4	1.98	36
Total Mining OPEX	**15.9**	**158.6**	**5.56**	**100**

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 as part of UGP operational readiness plans. These roles become part of the Mine General labour OPEX.

21.2.2.2.2 Owner's UG Mining Equipment

Owner's UG mining equipment usage costs are based on Lucara developed maintenance schedules and estimates prepared by the operational readiness team and verified by the QP. Cost basis is dependent on the mining equipment, in general:

- Fixed equipment costs are estimated in annual periods based on historical site data, OEM guidance, or Lucara estimations. Fixed equipment costs include service costs and wear components, as well as major scheduled work such as rope replacements or jaw crusher liner changeouts; and

- Mobile equipment costs are estimated in monthly periods; equipment unit costs are back calculated based on operating hours (US$/NOH) and verified against OEM provided costs and QP experience. Mobile equipment costs include service kits, maintenance parts, tires, oils, buckets, and other ground engaging tools.

Equipment costs are primarily associated with the mine maintenance activity and not the specific activities for load and haul, crushing and hoisting, or mine general. This distribution of costs is to align with how budgets and estimates have been established by the Lucara operational readiness team. Mine General equipment costs are primarily associated with site facility services equipment, IT, etc., and exclude all equipment falling under the responsibilities of the Mine Maintenance department.

Mobile equipment used and maintained by the contractor is included directly in the contract rates and excluded from this Owner's UG summary.

Operating costs for mobile equipment owned by Lucara do not account for any rental or financing charges against the equipment; all of these costs have been capitalized. Equipment mid-life overhauls are included in the sustaining capital costs at 60% the cost of new. End of life equipment replacements have also been included in the sustaining capital costs.

Table 21-27: Owner's UG Equipment Cost Summary

Owner's Equipment OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Mine Maintenance	5.8	58.4	2.05	96
Mine General	0.2	2.2	0.08	4
Total Mining OPEX	**6.1**	**60.6**	**2.12**	**100**

A summary by work area of the Owner's UG equipment costs outlines annual and unit rate operating costs in Table 21-28.

Table 21-28: Owner's UG Equipment Annual and Unit Rate Costs

Working Area	Average Annual	Unit Cost	Comment
	US$M	US$ / NOH	
Extraction Level LHDs	1.6	72	5 Operating Units - LH621s and LH517s.
Mobile Equipment - Auxiliary	1.4	47	Blended Rate of 17 units.
Shaft and Hoisting	0.5	-	Rope replacements and conveyance equipment.
Crusher and Conveyor	0.4	-	Rockbreaker, crusher, VGF and conveyor costs.
Dewatering System	0.1	-	6 operating pumps excluding stormwater inflows.
Ventilation	0.3	-	BAC and main fans.
Electrical and C&I	0.7	-	Includes motors and distribution system.
Various	1.1	-	Surface equipment, light vehicles, equipment hire.
Total Equipment OPEX	**6.1**		

21.2.2.2.3 UG Mining Materials

Owner's UG mining materials costs are based on Lucara developed estimates prepared by the operational readiness team and verified by the QP. UG mining material estimates are typically allowances for consumables in the mine maintenance and mine general activities based on historical site budgets.

UG mining materials costs include allowances for:

- Piping, valves, hoses and fittings;
- Electrical bulks, switches, splice kits, connectors and glands;
- Construction and repair materials;
- Lubricants, fluids, paint, and gases; and
- Maintenance tooling.

A summary of UG mining materials costs is provided in Table 21-29.

Table 21-29: UG Mining Materials Summary

Owner's Materials OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Mine Maintenance	0.6	6.1	0.21	58
Mine General	0.4	4.4	0.15	42
Total Mining OPEX	**1.0**	**10.4**	**0.37**	**100**

21.2.2.2.4 UG Explosives

UG explosives costs are based on first principal quantity estimates and unit rates provided by the on-site explosives supplier. Explosives are free-issued materials provided by the Owner's Team to the Contractor.

Table 21-30: UG Explosives Cost Summary

Explosives OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Lateral Development	0.5	1.1	0.04	12
Drill & Blast	4.1	8.2	0.29	88
Total Mining OPEX	**4.6**	**9.3**	**0.33**	**100**

21.2.2.2.5 UG Fuel Costs

UG mining fuel consumption has been estimated from contractor provided schedules, first principles operating hour estimates for Owner performed activities, and factors for surface and underground construction requirements. The fuel consumption estimate has been calibrated against current operating actuals. A 5% contingency has been added to the estimate to account for out of scope requirements.

Equipment fuel consumption rates have been sourced from OEMs or the list of CANMET-MMSL approved diesel engines for use in UG mines. Engine load and utilization factors by varying types of equipment are applied to CANMET rates to estimate fuel consumption during operations. All engines are rated tier 3 as there is currently no access to ultra-low sulfur diesel fuel in Botswana to support the operation of tier 4 or 5 motors.

The unit fuel price used in the estimate is US$1.08/litre, inclusive of delivery to site.

Fuel is a free-issued material provided by the Owner's Team to the Contractor.

Table 21-31: UG Fuel Cost Summary

Fuel OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Lateral Development	0.1	0.9	0.03	4
Drill & Blast	0.0	0.0	0.00	0
Load and Haul	0.7	7.1	0.25	34
Crushing and Hoisting	-	-	-	0
Surface Haulage	0.8	7.7	0.27	37
Mine Maintenance	-	-	-	0
Mine General	0.5	5.0	0.17	24
Total Mining OPEX	**2.1**	**20.7**	**0.73**	**100**

21.2.2.2.6 UG Power Consumption

Electrical power consumption has been based on a site load list and operating schedule which accounts for absorbed power factors, utilization factors and seasonal operating conditions. The power consumption estimate has been calibrated against current operating actuals.

UG mining power includes the power consumption for the working areas established as part of the UGP such as the shafts and ventilation system. Power costs associated with the existing plant site are summarised separately as part of Process operating costs under the 'Other Power Costs' category.

Power costs are based on a budgetary rate of US$0.076/kWh with a monthly demand rate of US$22.32/kW at peak demand. Demand charges are based on the peak demand, which are higher than the stated averages.

Table 21-32: UG Power Cost Summary

Power OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Lateral Development	0.1	0.6	0.02	1
Drill & Blast	0.1	0.1	0.00	0
Load and Haul	-	-	-	0
Crushing and Hoisting	1.8	18.3	0.64	31
Surface Haulage	-	-	-	0
Mine Maintenance	-	-	-	0
Mine General	4.0	40.1	1.40	68

Power OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Total Mining OPEX	**5.9**	**59.1**	**2.07**	**100**

A power consumption summary is provided in Table 21-33.

Table 21-33: UG Power Consumption

Power Consumption	Average Peak Demand (kW)	Average Annual (MWh)	Unit Cost per tonne Processed (kWh/t)	Weighting (%)
Mobile Equipment	91	608	0.21	1
Shaft and Hoisting	1465	10,100	3.54	23
Crusher and Conveyor	516	3,559	1.25	8
Dewatering	1058	7,309	2.56	17
Ventilation	1593	10,945	3.84	25
Mine Air Cooling	819	5,640	1.98	13
Maintenance Facilities	23	161	0.06	0
Compressed Air	148	1,013	0.36	2
Surface Water Management	266	1,846	0.65	4
Misc Other	426	2,938	1.03	7
Total Load	**6405**	**44,118**	**15.48**	**100**

21.2.2.2.7 Other

Other UG mining costs have been estimated by Lucara's operational readiness team and verified by the QP. These costs have all been categorized under the Mine General activity and include:

- Access road maintenance and staff transportation;
- Training and education costs; professional membership fees;
- Admin expenses, site meals, office supplies, and guest accommodation;
- Underground health and safety, audits, and medical examinations;
- Environmental monitoring fees and services;

- Annual software licensing and software support; and
- Survey support and technical equipment calibrations.

Table 21-34: UG Owner's Other Cost Summary

Owner's Other OPEX	Average Annual (US$M)	Life of Mine (US$M)	Unit Cost per tonne Processed (US$/t)	Weighting (%)
Mine General	4.1	41.1	1.44	100
Total Mining OPEX	**4.1**	**41.1**	**1.44**	**100**

21.2.3 Processing Operating Cost Estimate

The process plant and site infrastructure costs are based on the existing plant's yearly operating budget provided by Lucara Botswana and include the following:

- Costs to manage and operate the process plant, CRD and FRD facilities, and water treatment facilities;
- Process plant site power including site facilities established prior to the UGP; and
- Engineering labour for site facilities excluding 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 21-35.

Table 21-35: Processing OPEX

Process Operating Costs	Average Annual	Life of Mine	tonnes Processed	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Process Costs	24.7	345.3	37.3	9.3	78
Other Power Costs	7.1	99.2	37.3	2.7	22
Total	**31.8**	**444.5**	**37.3**	**11.9**	**100**

21.2.4 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:

- Human resources and site administration;
- Site finance, procurement, and IT;
- Safety, health, and environment;
- 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 21-36.

Table 21-36: G&A OPEX

G&A Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
G&A	18.5	259.1	37.3	6.9	100
Total	**18.5**	**259.1**	**37.3**	**6.9**	**100**

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.

21.2.5 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. Out of country corporate costs associated with Lucara Diamond have been excluded from the operating costs.

A summary of costs is provided in Table 21-37.

Table 21-37: LOM Sales and Corporate Cost

Sales Operating Costs	Average Annual[1]	Life of Mine	Tonnes Processed[2]	Unit Cost per tonne Processed	Weighting
	US$M	US$M	Mt	US$/t	%
Cost of Sales	3.6	49.8	37.3	1.3	34
Corporate Charges	6.8	95.0	37.3	2.5	66
Total	**10.3**	**144.8**	**37.3**	**3.9**	**100**

22 ECONOMIC ANALYSIS

An engineering economic model was developed to estimate the annual cash flows and sensitivities of the Project. After-tax estimates of project values were developed and are likely to approximate the true investment value. It must be noted 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.

The economic model was prepared by Lucara Diamond and verified by the QP. Lucara Diamond has performed the tax estimates based on their >10 years of operating experience in Botswana.

Univariate sensitivity analyses were performed for variations in diamond prices, operating costs, and capital costs, to determine their relative importance as project value drivers.

This Report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. No provision is made, or contingency allowed for external project risk, unidentified changes to the scope or work, general project escalation, or foreign exchange fluctuations, some of which may cause actual results to differ materially from those presented. Limited contingency allowance for internal project risk has been allowed as defined in the Project Change Log. A summary of risks that apply to Lucara Diamond, KDM, and the UGP are described in Section 25.2.

The estimates of capital and operating costs have been developed specifically for this Project and are summarised in Section 21 of this Report (presented in 2025 US dollars). The economic analysis has been performed with no escalation or inflation (constant dollar basis).

22.1 Summary of Results

The summary of the mine plan and payable diamonds produced is outlined in Table 22-1. 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 22-1: LOM Summary of Economic Results

Parameter	Unit	Value
Ore Processed	Mt	37.3
Mill Average Annual Production	Mt	2.85
Average Processing Grade	cpht	12.2
Diamonds Recovered	k ct	4,547
Recovery	%	100.0
Net Revenue (after royalties)	US$M	2,991.9
Operating Costs	US$M	1,315.4

Parameter	Unit	Value
Income Taxes and Working Capital	US$M	316.5
Operating Cash Flow	US$M	1,360.0
Pre-Production Capital Cost (excluding incurred costs to September 30, 2025)	US$M	290.8
Sustaining Capital Cost and Closure	US$M	234.0
Contingency	US$M	59.3
After-tax Free Cash Flow	US$M	775.9
Net Present Value @ 8%	US$M	432.1

Other economic factors include the following:

- Discount rate of 8%;
- Nominal 2025 US$ dollars;
- Revenues, costs, and 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;
- Lucara Botswana corporate costs are included in all economic results;
- Canada corporate (Lucara Diamond Corp.) costs are not included in the base case economic model results;
- Working capital is included;
- Debt financing costs are excluded from the analysis; and
- The model excludes all incurred costs up to the base date of September 30, 2025.

22.2 Assumptions

Table 22-2 outlines the exchange rate assumptions used in the economic analysis. The BWP:USD exchange rate of 14.0 has been provided by Lucara and verified by the QP, based on current transaction rates. Currently, there is a large spread between the posted Bank of Botswana exchange rates and actual transaction rates, as the Bank of Botswana attempts to limit local banks from buying up USD reserves.

Table 22-2: Exchange Rate Assumptions

US$	Exchange Rates	Currency
1 USD =	14.0	BWP
	17.5	ZAR

Table 22-3 and Table 22-4 outline the diamond prices 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 and sales (Section 19).

Table 22-3: Baseline Diamond Prices

Unit	Unit	2025 FS
North	US$/ct	155
Centre	US$/ct	210
EM/PK(S)	US$/ct	695
M/PK(S)	US$/ct	520
Stockpiles	US$/ct	442

Table 22-4: Legacy Stone (>US$5 million) Values

Unit	Occurrence Stones / Mtonne	Diamond Price US$ M / Stone	Revenue US$ M / Mtonne[1]
EM/PK(S)	3.34	12.9	43.0
M/PK(S)	1.45	9.5	13.7

Notes:
1. Undiluted tonnes.

Efforts have been made to provide realistic estimates for exchange rates and diamond prices based on historical performance and sales information. It should be noted that exchange rates and diamond prices are based on many complex factors and there are no reliable long-term predictive tools.

22.3 Royalties

KDM is subject to a royalty payable to the Botswana Government of 10% of all sales. Since royalties are applied on the HB final sales prices, the effective royalty payment with the HB agreement in place is approximately 11.2% The government authorization of HB contract expires in December 2032. Estimated royalty payments amount to US$362.4 million over the remaining LOM.

22.4 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 Diamond and contains the following assumptions:

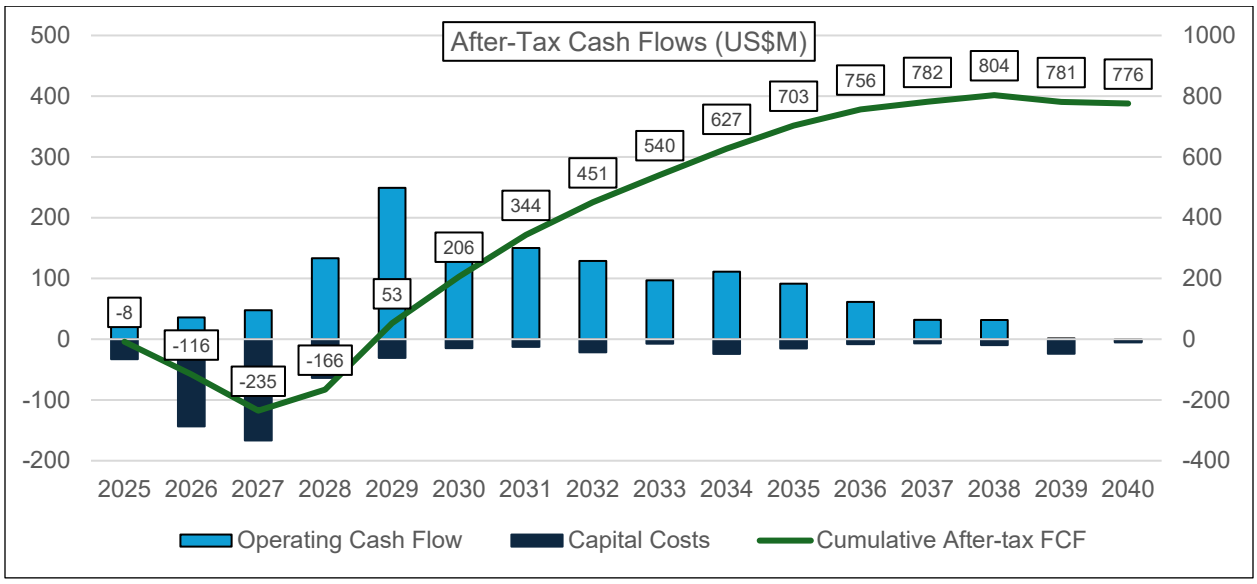
- Income Tax: Annual tax rate = 70 – 1500/x:
  - Where x is the profitability ratio, given by taxable income as a percentage of gross income;
  - Where the calculated rate shall not be less than the company rate of 22%; and
  - Net Losses, incurred in years of high capital expenditures, can be deferred to future years to offset tax liabilities.
- VAT modelled with a three-month delay on refunds have been incorporated into Working Capital requirements; and
- Withholding taxes on foreign consulting services are not expected and estimated at US$0 in the economic model.

Interest expense is generally tax deductible; however, financing costs have been excluded from the income tax calculations, and the income tax forecast has been prepared on a fully unlevered basis. As a result, forecast income taxes may be higher than taxes ultimately payable under a leveraged financing structure. Total corporate income taxes for the remaining Project LOM are estimated at the amount of US$313.2 million.

22.5 Results

The KDM LOM, including the development of the UGP, has an after-tax net present value using an 8% discount rate ($NPV_{8\%}$) of US$432.1 million using the cost assumptions described in Section 22.2. Figure 22-1 shows the projected KDM cash flows, and Table 22-5 summarises the economic results for the mine.

Figure 22-1: After-Tax Cash Flows



The after-tax break-even average diamond price and legacy diamond value is approximately 67% of the base case prices, based on the LOM plan presented herein. This is the diamond price and legacy diamond value at which the Project $NPV_{8\%}$ is zero.

The LOM economic model does not calculate a meaningful Internal Rate of Return (IRR) for two key reasons:

- Incurred costs of US$436.0 million on the project are not considered in IRR calculations; and
- Project capital costs are partially offset by operating revenue during the years they are incurred.

Project economic results are reported in Table 22-5 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 22-5: After-Tax Economic Results - LOM Model

Parameter	Unit	After-tax Results
$NPV_{8\%}$ excluding Canadian corporate costs	US$M	432.1
$NPV_{6\%}$ excluding Canadian corporate costs	US$M	505.5
$NPV_{8\%}$ including Canadian corporate costs	US$M	368.5
$NPV_{6\%}$ including Canadian corporate costs	US$M	434.7

22.6 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. The Project showed the least sensitivity to capital costs. Table 22-6 show the results of the sensitivity tests.

Table 22-6: Sensitivity Results (After-Tax NPV @ 8%)

Variable	After-tax $NPV_{8\%}$ (US$M)				
	-20% Variance	-10% Variance	Base	+10% Variance	+20% Variance
Diamond Price	248.8	342.6	432.1	518.0	602.4
Mining Cost	457.5	445.0		418.8	404.7
Processing Cost	454.6	443.6		420.6	408.8
All Operating Costs	512.1	473.1		388.0	342.2
Upfront CAPEX	474.3	453.2		411.3	390.5
Sustaining CAPEX	446.3	439.2		425.0	417.9
All capital costs	488.1	460.3		404.1	376.1

The economic model summary used for the LOM mine is summarised in Table 22-7.

Table 22-7: LOM Annual Cash Flow

	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
PRICE																		
Diamond Price																		
North	US$/ct	155	155	155	155	155	155	155	155	155	171 [1]	171	171	171	171	171	171	171
Centre	US$/ct	210	210	210	210	210	210	210	210	210	232 [1]	232	232	232	232	232	232	232
South - EMPKS	US$/ct	695	695	695	695	695	695	695	695	695	768 [1]	768	768	768	768	768	768	768
South - M/PK(S)	US$/ct	520	520	520	520	520	520	520	520	520	575 [1]	575	575	575	575	575	575	575
SP: Life of Mine	US$/ct	442	442	442	442	442	442	442	442	442	488 [1]	488	488	488	488	488	488	488
Legacy Diamonds																		
South - EMPKS	US$M/Stone	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9	12.9
South - M/PK(S)	US$M/Stone	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5
MINE PRODUCTION																		
Waste Mined	k tonnes	**836**	**20**	**228**	**470**	**118**	0	0	0	0	0	0	0	0	0	0	0	0
Underground	k tonnes	836	20	228	470	118	0	0	0	0	0	0	0	0	0	0	0	0
Ore Mined	k tonnes	**29,644**	**529**	**608**	**516**	**2,448**	**2,783**	**2,794**	**2,850**	**2,858**	**2,850**	**2,850**	**2,850**	**2,858**	**2,850**	0	0	0
Open Pit	k tonnes	1,125	529	596	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Underground	k tonnes	28,519	0	12	516	2,448	2,783	2,794	2,850	2,858	2,850	2,850	2,850	2,858	2,850	0	0	0
Ore Grade	**cpht**	**12.8**	**12.3**	**11.6**	**14.9**	**17.4**	**17.6**	**17.6**	**15.6**	**13.5**	**12.6**	**11.7**	**10.0**	**7.6**	**5.6**	0.0	0.0	0.0
Open Pit	cpht	11.9	12.3	11.5	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
Underground	cpht	12.9	0.0	15.0	14.9	17.4	17.6	17.6	15.6	13.5	12.6	11.7	10.0	7.6	5.6	0.0	0.0	0.0
Contained Carats	**k carats**	**3,803**	**65**	**70**	**77**	**425**	**489**	**493**	**444**	**385**	**358**	**334**	**284**	**218**	**161**	0	0	0
Open Pit	k carats	134	65	69	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Underground	k carats	3,669	0	2	77	425	489	493	444	385	358	334	284	218	161	0	0	0
STOCKPILE MANAGEMENT																		
Opening balance	**k tonnes**	**7,634**	**7,634**	**7,472**	**5,229**	**2,895**	**2,486**	**2,419**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**0**	**0**	
Stockpile added	k tonnes	*0*	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0.0	0.0
Stockpile removed	k tonnes	*-7,634*	-163	-2,242	-2,334	-410	-67	-56	0	0	0	0	0	0	0	-2,363	0.0	0.0
Closing balance	**k tonnes**	**0**	**7,472**	**5,229**	**2,895**	**2,486**	**2,419**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**2,363**	**0**	**0**	**0**
MILL FEED																		
Ore Processed	k tonnes	37,278	692	2,850	2,850	2,858	2,850	2,850	2,850	2,858	2,850	2,850	2,850	2,858	2,850	2,363	0.0	0.0
	k t/d	7.8	7.5	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	0.0	0.0
Milled Grade	cpht	12.2	12.3	12.7	11.6	15.6	17.3	17.4	15.6	13.5	12.6	11.7	10.0	7.6	5.6	6.4	0.0	0.0
Contained Carats	k carats	4,547	85	363	330	446	492	495	444	385	358	334	284	218	161	152	0	0
Recovery Rate	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Recovered Carats	**k carats**	**4,547**	**85**	**363**	**330**	**446**	**492**	**495**	**444**	**385**	**358**	**334**	**284**	**218**	**161**	**152**	0	0
North	k carats	51	0	0	0	0	0	0	0	0	0	0	0	0	0	51	0	0
Centre	k carats	408	15	202	191	0	0	0	0	0	0	0	0	0	0	0	0	0
South - EMPKS	k carats	2,459	21	11	75	411	460	431	296	175	125	122	118	114	101	0	0	0
South - M/PK(S)	k carats	1,432	42	150	2	13	29	62	148	211	234	213	166	104	60	0	0	0
SP: Life of Mine	k carats	198	8	0	62	21	3	3	0	0	0	0	0	0	0	101	0	0
Legacy Diamonds	**#**	**57**	0	**1**	**1**	**7**	**7**	**8**	**7**	**6**	**5**	**5**	**4**	**3**	**3**	0	0	0

	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
South - EMPKS	#	40	0	0	1	7	7	7	5	3	2	2	2	2	2	0	0	0
South - M/PK(S)	#	17	0	1	0	0	0	1	2	3	3	3	2	1	1	0	0	0
REVENUE																		
Diamond Revenue (excl. legacy diamonds)	US$M	2,724.7	42.7	127.9	120.7	302.1	336.3	332.8	282.4	230.9	230.0	215.7	186.1	147.1	112.0	58.1	0.0	0.0
Legacy Diamond Revenue	US$M	629.6	0.0	29.3	11.0	76.7	76.7	84.7	70.9	57.0	54.1	54.1	44.7	35.2	35.2	0.0	0.0	0.0
Royalties	US$M	-362.4	-4.8	-17.7	-14.8	-42.6	-46.4	-46.9	-39.7	-32.3	-28.4	-27.0	-23.1	-18.2	-14.7	-5.8	0.0	0.0
Cost of Sales	US$M	-49.8	-1.0	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-3.9	-1.6	0.0	0.0
Corporate Charges (Botswana)	US$M	-95.0	-1.9	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-7.5	-3.1	0.0	0.0
Net Revenue	**US$M**	**2,847.1**	**35.1**	**128.1**	**105.4**	**324.8**	**355.1**	**359.1**	**302.2**	**244.1**	**244.3**	**231.4**	**196.3**	**152.6**	**121.0**	**47.5**	**0.0**	**0.0**
OPERATING COSTS																		
On-site Operating Costs	**US$M**	**-1,170.6**	**-20.3**	**-73.0**	**-61.0**	**-115.7**	**-101.6**	**-95.3**	**-95.2**	**-94.7**	**-95.0**	**-94.5**	**-95.0**	**-94.5**	**-91.7**	**-43.0**	**0.0**	**0.0**
Open Pit Mining Costs	*US$M*	-20.3	-6.7	-13.6	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
Underground Mining Costs	*US$M*	-427.3	0.0	0.0	0.0	-59.9	-47.1	-40.8	-40.8	-40.2	-40.6	-40.1	-40.6	-40.1	-37.3	0.0	0.0	0.0
Stockpile Rehandle Costs	*US$M*	-19.3	0.0	-5.0	-6.6	-1.4	-0.1	-0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-6.2	0.0	0.0
Process Costs	*US$M*	-345.3	-6.6	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-26.4	-22.0	0.0	0.0
Other Power Costs	*US$M*	-99.2	-1.9	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-7.6	-6.3	0.0	0.0
G&A	*US$M*	-259.1	-5.1	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-20.5	-8.5	0.0	0.0
Off-site Operating Costs	**US$M**	**-316.5**	**9.6**	**-19.2**	**3.2**	**-75.8**	**-4.5**	**-95.6**	**-56.8**	**-20.7**	**-52.3**	**-25.9**	**-9.9**	**3.3**	**2.7**	**27.1**	**1.4**	**-3.1**
Corporate Income Tax	*US$M*	-313.2	0.0	0.0	0.0	0.0	0.0	-89.6	-76.1	-39.2	-49.7	-31.5	-18.6	-8.6	0.0	0.0	0.0	0.0
Working Capital and Others	*US$M*	-3.3	9.6	-19.2	3.2	-75.8	-4.5	-6.0	19.3	18.5	-2.6	5.6	8.6	12.0	2.7	27.1	1.4	-3.1
Total Cash Operating Costs	**US$M**	**-1,487.0**	**-10.7**	**-92.2**	**-57.8**	**-191.5**	**-106.1**	**-190.9**	**-152.0**	**-115.4**	**-147.3**	**-120.4**	**-104.9**	**-91.2**	**-89.0**	**-15.9**	**1.4**	**-3.1**
OPERATING CASH FLOW	**US$M**	**1,360.0**	**24.4**	**35.9**	**47.6**	**133.3**	**249.0**	**168.2**	**150.2**	**128.7**	**97.0**	**111.0**	**91.3**	**61.5**	**32.0**	**31.6**	**1.4**	**-3.1**
CAPITAL COSTS																		
Pre-Production Capex	**US$M**	**-290.8**	**-28.9**	**-110.0**	**-131.2**	**-20.7**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mining	*US$M*	-183.9	-22.6	-73.1	-78.5	-9.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site Development	*US$M*	-3.8	0.0	-0.4	-3.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Process Plant	*US$M*	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
Tailings	*US$M*	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
On-site Infrastructure	*US$M*	-0.2	-0.2	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
Buildings and Facilities	*US$M*	-0.7	-0.1	-0.3	-0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Off-site Infrastructure	*US$M*	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
Project Indirects	*US$M*	-10.6	-0.7	-3.7	-4.7	-1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Owners Costs	*US$M*	-91.6	-5.3	-32.4	-44.3	-9.6	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 Capex	**US$M**	**-234.0**	**-1.8**	**-11.5**	**-9.3**	**-38.7**	**-29.7**	**-14.0**	**-12.0**	**-20.8**	**-7.4**	**-23.4**	**-14.9**	**-8.1**	**-6.8**	**-9.7**	**-23.9**	**-2.0**
Mining	*US$M*	*-69.6*	0.0	0.0	0.0	-24.8	-9.5	-7.2	-5.2	-9.0	-0.6	-8.0	-3.9	-1.3	0.0	0.0	0.0	0.0
Site General	*US$M*	*-92.2*	-1.5	-9.0	-7.5	-7.2	-6.8	-6.8	-6.8	-6.8	-6.8	-6.8	-6.8	-6.8	-6.8	-5.7	0.0	0.0
Tailings	*US$M*	*-42.4*	-0.3	-2.6	-1.7	-6.7	-13.4	0.0	0.0	-5.0	0.0	-8.6	-4.1	0.0	0.0	0.0	0.0	0.0
Closure	*US$M*	*-29.9*	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-4.0	-23.9	-2.0
Contingencies	**US$M**	**-59.3**	**-2.0**	**-22.1**	**-26.4**	**-4.4**	**-1.0**	**-0.7**	**-0.5**	**-0.9**	**-0.1**	**-0.8**	**-0.4**	**-0.1**	0.0	0.0	0.0	0.0
Pre-Production Contingency	*US$M*	*-52.4*	-2.0	-22.1	-26.4	-1.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Contingency	*US$M*	*-7.0*	0.0	0.0	0.0	-2.5	-1.0	-0.7	-0.5	-0.9	-0.1	-0.8	-0.4	-0.1	0.0	0.0	0.0	0.0

	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Total Capital Costs	**US$M**	**-584.1**	**-32.8**	**-143.6**	**-166.8**	**-63.8**	**-30.7**	**-14.7**	**-12.5**	**-21.7**	**-7.4**	**-24.2**	**-15.3**	**-8.3**	**-6.8**	**-9.7**	**-23.9**	**-2.0**
AFTER-TAX FREE CASH FLOW	**US$M**	**775.9**	**-8.4**	**-107.7**	**-119.2**	**69.5**	**218.4**	**153.5**	**137.6**	**107.0**	**89.5**	**86.9**	**76.0**	**53.2**	**25.2**	**22.0**	**-22.5**	**-5.1**

Notes:

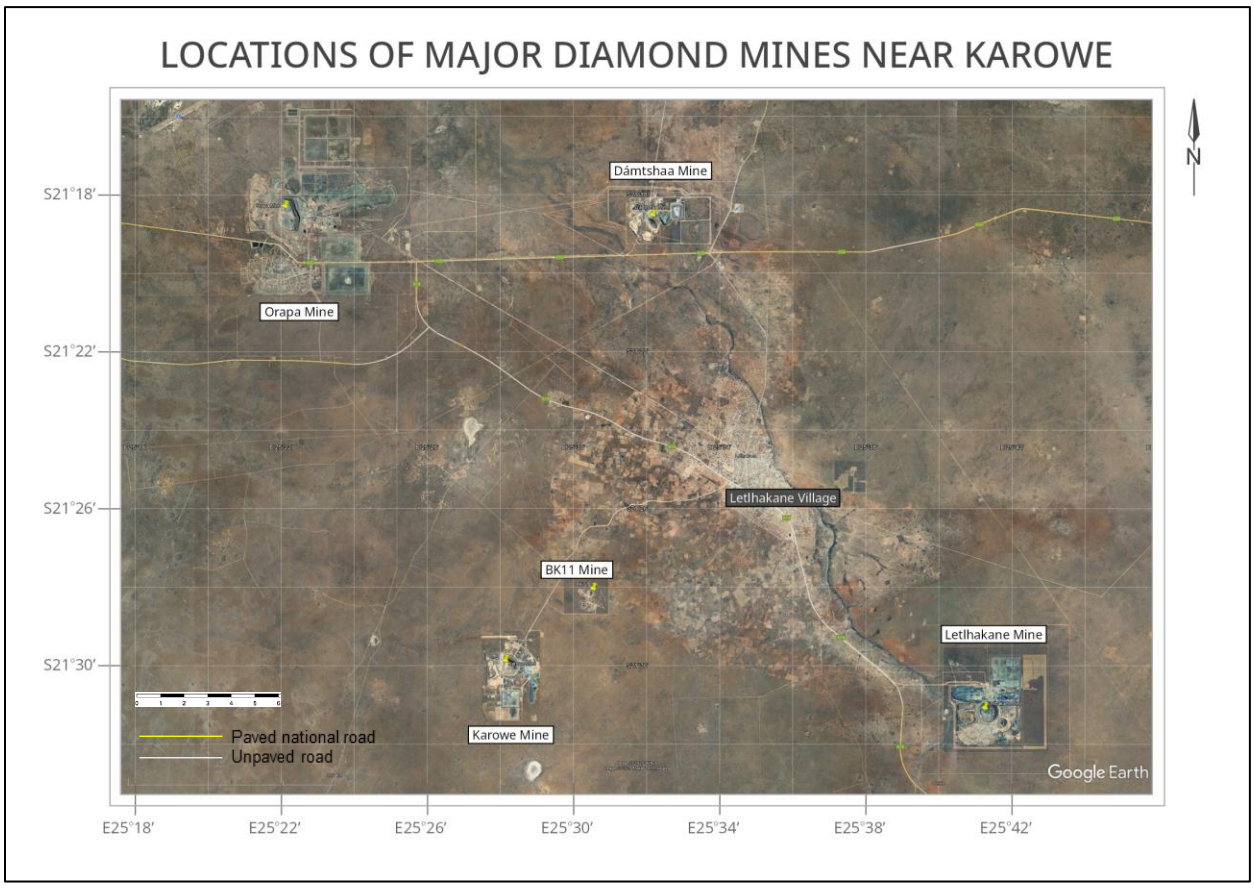
1. Diamond Price increased in 2033 after expiration of current HB NDSA as outlined in Section 19.2.

23 ADJACENT PROPERTIES

There are numerous kimberlite pipes within a 50 km radius of KDM as well as several operating and past-producing diamond mines. Debswana Diamond Company, an equal partnership of the De Beers Group and the Government of the Republic of Botswana, owns three mines in the area, the largest and by far the most significant of which is the Orapa Mine. Anglo American owns 85% of the De Beers Group. The other two Debswana mines, Damtshaa and Letlhakane, have ended open pit operations after 18 and 42 years of production, respectively. Damtshaa Mine was placed on care and maintenance in 2021 and remains dormant at this time. Open-pit operations at Letlhakane Mine ceased as planned in 2017. The plant for processing tailings, which will extend the life of the operation to 2043, is now operational and has a capacity of up to 800,000 ct per annum. A summary of the Debswana owned mines can be found in Table 23-1. The QP has been unable to verify the information presented in the table and notes that the information is not necessarily indicative of the mineralization on the KDM property.

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 23-1 shows the location of local producing and past-producing mines.

Figure 23-1: Locations of Major Diamond Mines Near KDM



Source: Google Earth

Table 23-1: Summary Information for the nearby Debswana-owned Orapa, Letlhakane and Damtshaa Mines

Parameter	Unit	Orapa		Letlhakane		Damtshaa
Mining Method		**OP**	**TMR & ORT***	**OP**	**TMR & ORT***	**OP**
M+I Resource Tonnes**	Mt	271.3	189.3	22.3	0.0	25.2
M+I Resource Grade**	cpht	98.4	67.3	31.7	6,644	21.8
Inf Resource Tonnes**	Mt	75.7	-	18.7	42.5	26.6
Inf Resource Grade**	cpht	85.6	-	27.8	26.9	24.1
Reserve Tonnes	Mt	85.7			25.2	-
Reserve Grade	cpht	150.4			21.6	-
Operating Life	years	15			19	
Status		Operating	Project	Care and maintenance	Operating	Care and maintenance

Notes:

The QP has been unable to verify the information contained in this table. The information is not necessarily indicative of the mineralization on the KDM property that is the subject of this Report.

*TMR = Tailings Mineral Resources, ORT – Old Recovery Tailings

** Resources are reported as additional to Reserves

Orapa OP Mineral Resources and Reserves BCO = 1.65 mm

Letlhakane TMR and ORT BCO = 1.15 mm

Tonnes values reported as 0.0 represent estimates less than 0.05 Mt

Source: Anglo American Ore Reserves and Mineral Resources Report (2024)

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Management Plans

KDM implements a comprehensive set of management plans designed specifically to ensure the safe and efficient delivery of the Project. These plans undergo regular review and are updated when necessary to maintain a current framework for organizing engineering, procurement, and construction activities.

While the Management Plans serve as a guide for Project execution, ongoing planning will continuously assess alternative strategies and identify opportunities to enhance overall value.

Table 24-1: Management Plans

Karowe Underground Project Management Plans	
Project Execution Plan	Construction Execution Plan
Document Control Plan	Communication Plan
Procurement Management Plan	Change Management Plan
Contract Management Plan	Logistics Execution Plan
Material Equipment Management Procedure	Emergency Response Plan
Project Specific Safety Plan	Mine Planning Management Plan
Ground Control Management Plan	Ventilation Management Plan
Underground Blast Management Plan	Quality Management Plan
Commissioning Procedure	Traffic Management Plan
Environmental Plan	Project Controls Execution Plan
Schedule Execution Plan	Asset Management Plan
Business Continuity Plan	Cave Management Plan
Ground Water Management Plan	Risk Management Plan
Resource Management Plan	

Source: Lucara (2025)

24.1.1 Project Development Schedule

The overall development period for the Project is estimated to take approximately eight years from the start of detailed engineering (Jan 2020) to reaching commercial production (Feb 2028). Forward looking schedules do not include contingency for unplanned delays.

As of the effective date of this FS, mine construction is underway, and the following critical path items remain to reach project completion:

- Vent Shaft Sinking and lining (Shaft sunk to completion December 2025, with lining to be completed in 2026);
- Shaft equipping, and skip loading infrastructure;
- UG development of drifts, raises, and extraction drawpoints;
- Installation of primary ventilation fans and dewatering systems;
- Installation of crushers, conveyors, bins, and chutes; and
- Production drilling of drawbells, undercuts, and longhole stopes.

24.1.2 Development Milestones

A level 1 schedule as shown in Table 24-2 illustrates the UGP development milestones.

Table 24-2: KDM UG Development Schedule

Description	2025	2026		2027		2028
	H2	H1	H2	H1	H2	H1
OPEN PIT						
OPEN PIT OPERATIONS						
SHAFTS						
PROD SHAFT - EQUIP and COMMISSION						
PROD SHAFT - SKIP HOISTING						
VENT SHAFT - SINKING						
VENT SHAFT - EQUIP and COMMISSION						
DEVELOPMENT						
670 LEVEL						
470 LEVEL						
405 LEVEL						
310 LEVEL						
285 LEVEL						
INFRASTRUCTURE						
MAIN FANS						
PUMP ROOM						

Description	2025	2026		2027		2028
	H2	H1	H2	H1	H2	H1
FINE ORE BIN						
CRUSHER						
CONVEYOR						
WORKSHOP						
UGP PRODUCTION						
DRAWBELLS						
UNDERCUT						
STOPING						
PRODUCTION ACHIEVED (75%)						

In 2026 the V/S will finish sinking and both shafts will be equipped, commissioned, and handed over for owner operations. Underground contractors will mobilize to commence lateral development, raise boring, and installation of underground infrastructure with a focus on the extraction horizon, skip loading level, and undercut. Primary fans will be installed on surface, and all temporary sinking infrastructure will be demobilized from site. First ore will be achieved at the extraction horizon and major infrastructure chambers will be excavated including crusher, conveyor, and fine ore bins.

In 2027 development will continue at the extraction horizon to support construction of permanent magazines, pump stations, refuge chambers, and workshops. Drill and blast horizons will be established to support long hole stoping. Underground material handling systems will be commissioned, which will trigger the start of longhole production drilling and blasting of the undercut and drawbells.

In 2028 all planned lateral development will be completed and production drill and blast will ramp up to full production. All production blasts will be drilled off and cave instrumentation installed.

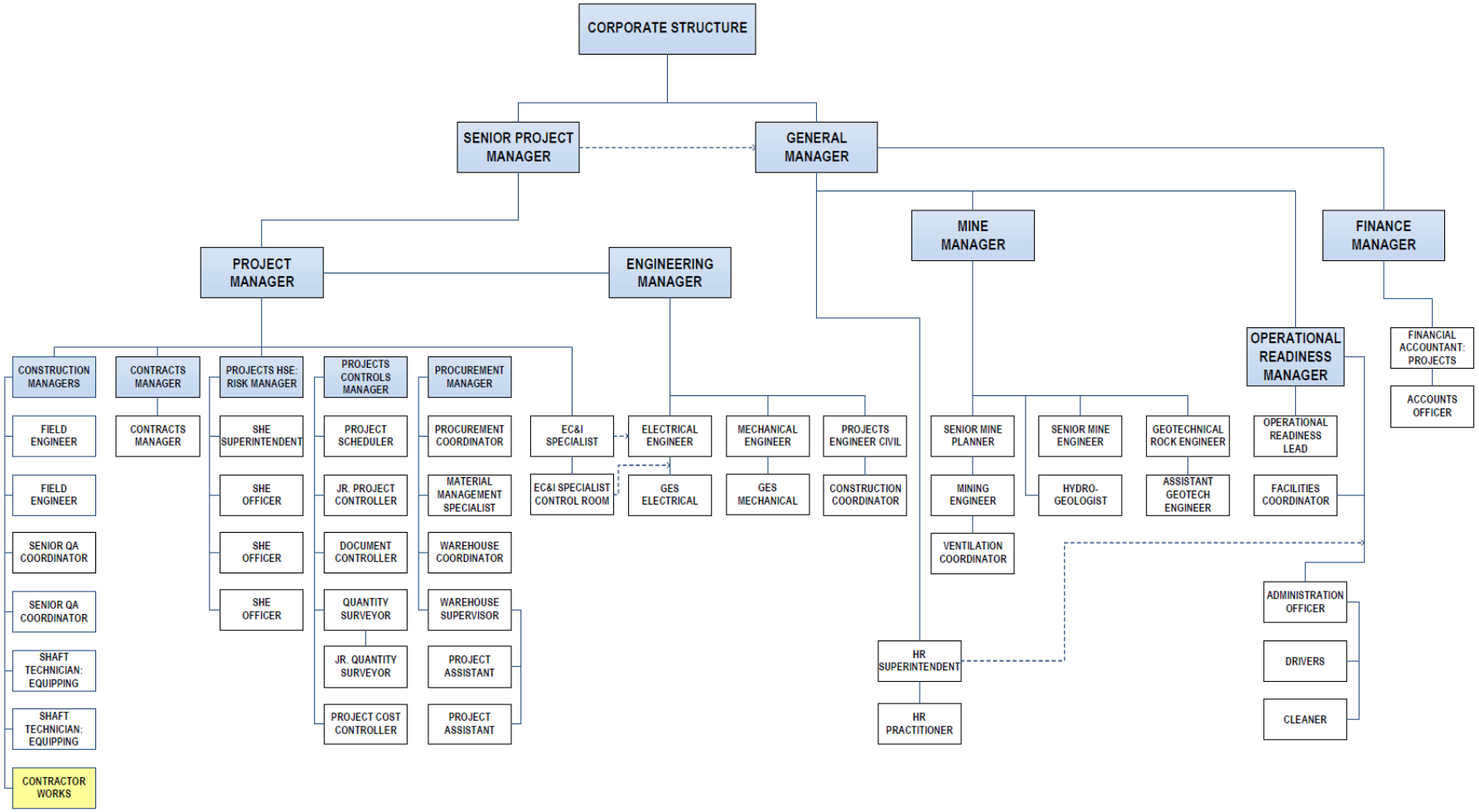
Natural caving is expected to initiate in early 2029.

24.1.3 Project Management

The Project Management Team (PM Team) operates within Lucara Botswana and receives consulting support as necessary. The PM Team oversees and directs all engineering, procurement, and construction activities for the Project. Figure 24-1 presents the current project organization chart for both the engineering and construction phases of the Project.

The PM Team operates as an individual project group within Lucara Botswana's organizational structure and utilises many services caried out by the Lucara Botswana team, including management of site consumables (power, water, fuel, explosives), site security, environmental and permitting, and accounting.

Figure 24-1: Organizational Structure



Source: Lucara (2025)

25 INTERPRETATIONS AND CONCLUSIONS

This Feasibility Study provides the technical, geological, environmental, and economic basis for extending operations at KDM through UG development. This report is updated from the 2023 FS and incorporates the following significant modifications:

- Project construction progress to September 30, 2025;
- Revised economic model with updated exchange rates, diamond prices, incurred costs to date, and inclusion of Legacy stone values;
- Re-baselining the open pit, UGP, and processing facility production plans;
- Re-estimation of the project capital and operating cost projections;
- Geological model updates incorporating new pit mapping and drillhole data;
- Hydrogeological and geomechanical model updates;
- Mine design, method, and schedule updates;
- Advancement of detailed engineering designs; and
- Groundwater management on surface.

The Mine is situated in Botswana with established access, utilities, and site infrastructure. Environmental approvals, monitoring programs, groundwater-management measures, and closure planning remain active.

The Project transitions from open-pit depletion in 2026 to underground production in 2028, between which existing stockpiles will maintain mill feed to the existing plant.

Rock mass data, stress measurements, cave simulations, and groundwater modelling define excavation designs, support requirements, and water-management measures. Metallurgical and XRT testwork confirm compatibility with existing processing circuits.

The underground plan utilizes a single extraction level, multiple drilling horizons, and a defined sequence for stoping and caving. Ore will be crushed UG, hoisted to surface, and processed in the existing plant. Required underground and surface infrastructure including power, water systems, material handling, and tailings storage is in place or scheduled prior to production.

Pre-production capital totals US$779.2 M (incurred and remaining), with sustaining capital of US$240.9 M. Life-of-mine operating costs are estimated at US$1,315 M. Economic analysis results in an after-tax NPV of US$432.1 M at an 8% discount rate.

25.1 External Risks

Most mining projects have a large variety of risks that can be either managed at the mine site or at the owner's Corporate level (internal risks), or which are beyond the control of the site (external risks).

External risks to Lucara Diamond, KDM, and UGP 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;
- 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; and
- Risks associated with the production and increased consumer demand for synthetic gem-quality diamonds.

25.2 Internal Risks

The internal risks 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, risks are removed and/or additional risks are recognised, and mitigating actions completed or new mitigations identified.

Risk identification within the registers was based on direct interviews and inputs from the QPs: geotechnical, hydrogeology, mining, and CRD/FRD management. The FS Risk Register is presented in Table 25-1.

25.2.1 Risk Registers

Lucara maintains risk registers for the Project, some of which are established and closed out as the Project enters and completes new phases of work. The risk registers currently managed by Lucara include:

- Project Risk Register;
- Development Risk Register; and
- Equipping Risk Register.

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 25-1: Feasibility Study Risks Register: Main Project Risks (not necessarily in order of significance)

Risk Statement	Description/Cause/Consequence	Mitigation
Financing requirements	Lucara plans to fund the UGP at KDM with cash from operations and external financing, committing much of its revenue and cash flow to the project. If revenues or cash flows are insufficient, or if project costs rise, Lucara will need more capital, which may not be easily available or on favourable terms. Failure to secure additional funding could significantly harm Lucara's business, finances, and operations.	• Appropriate operational and Project controls are established as well as continual, focused identification and pursuit of optimizations and improvements by the site teams; and • Ongoing strong connection and communication with shareholders and Lenders.
Labour Supply	Difficulty in attracting and retaining skilled staff could negatively impact project schedule and cost for reasons including: • Limited local underground experienced labour leads to increased demand for unbudgeted expatriate participation; • Market competition and/or compensation budgets drive low interest in attracting suitably trained and experienced employees; and • Failure to secure or renew permits for expatriate labour; and • High turnover of personnel, leading to recruiting, quality, and training demands.	• The UGP to date has had a good 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; • Ongoing close communication with government on work permit needs; • Project labour budgets and contingencies are considered healthy; • Certain key functions, such as recruiting and training of technical services, are to be given a high priority and managed accordingly; and • The Project retains the option of supplementing the workforce with additional skilled labour and experts to support the work, should that be necessary.
Lateral Development	Failure to achieve planned monthly advance rates on critical path tasks will drive delays, increasing both cost, schedule, and revenue gaps. Potential causes include: • Overestimated productivity rates; • Current definition of ground conditions on the extraction and undercut levels is based on limited drilling. Changes in expected physical conditions (geotechnical, hydrogeological, atmospheric, or other) may occur, requiring additional support, dewatering, probing, or other activities within the cycle;	• The Project uses lateral development and construction rates which are considered reasonable and supported by multiple independent development contractor bids; • An allowance for probe drilling ahead of development faces has been included; • Dewatering drain holes are planned, to identify and depressurise groundwater ahead of mining; • Above, coupled with future surface drill programs, will enable further definition of the leached granite, contact zones, and major structures;

Risk Statement	Description/Cause/Consequence	Mitigation
	• Improperly specified, supplied, or maintained equipment, tools, or shaft conveyances; • Increased wear, maintenance, or replacement frequency of equipment or ground support due to corrosive nature of groundwater • Insufficient, undersized, or delayed supply or installation of services and material handling systems; • Leached granite has the potential for locally high short-duration water inflows; • Falls of ground in development headings (resulting from failure to comply with established ground support standards, or revision of ground support to suit ambient and evolving geotechnical conditions) would cause delays and additional remedial support; • Scope changes or additional hold points impacting the cycle; • Overestimated effective working time; and • Underestimated congestion, logistics, or precedent logic.	• Full implementation of the GCMP will mitigate fall-of-ground risks, and also include TARPs, SOPs, and QAQC of ground support installation; • Adoption of corrosion-resistant ground support in high-risk areas; • Training and real time monitoring to allow adaptive design of ground support; • Ground support is industry standard and applied to the mine design using an RMR block model, with attributes according to the GCMP, ensuring heavy support is applied where drillhole data or modelling identifies poor ground conditions; • Completion of future drill programs, coupled with geotechnical mapping during development, will identify any deviations from the plan and additional ground control responses designed; • Power and air supply systems are oversized for the development period and can accommodate unplanned increases in demand; • Working times account for shaft availability, and end of shift blasting, among other factors; • The FS includes updated materials handing simulations to examine congestion issues; and • Project schedule allows for two-week grouting campaigns for every drive entering the kimberlite.
Production Drill & Blast	Complications during execution of the proposed drill & blast method will result in increased cost and/or reduced throughput. Potential causes include: • Hole collapse or inaccuracies requiring excessive redrill; • Tight firing constraint results in misfires, freezing, or poor blasting performance requiring redrills, oversize, and/or delays; • Reliance on wireless detonators and the learning curve associated; • Cave initiation occurs earlier than expected, resulting in some longhole rings not being blasted (no longer safe for personnel to enter blasting areas);	• Drill lengths, patterns, and throughputs are within industry precedents for open stoping, but will be further refined by independent drill & blast experts during detailed design; • High-efficiency drill equipment is planned and appropriately specified for the proposed drill patterns, and performance monitored during execution; all blastholes will be downhole surveyed; • The mining method is supported by strong back-up of data from extensive drilling and geotechnical modelling, and multiple third-party reviews by leading drill and blast industry experts;

Risk Statement	Description/Cause/Consequence	Mitigation
	• Lack of adequate QAQC on blast design, implementation, and performance; and • Overestimation of bulk density (secondary fragmentation and recompaction) leads to increased blasting rate to meet production.	• Use of wireless detonation will enable multiple rings to be pre-charged, significantly reducing personnel exposure to the blasting areas; • Careful monitoring of airgap, muck pile, draw and blasthole stability will be implemented; and • Flexible mine design back-bone infrastructure allows for improvements to mine methods as necessary.
Cave Propagation	Failure of the cave to propagate as predicted may result in slower cave performance, delayed mine production, and increased operating costs. Potential causes include: • Chimneying of the cave along weak contacts or features within the kimberlite, or along the fracture corridor, could change the course of caving direction and sterilize portions of the orebody, or result in earlier dilution entry. It can also jeopardise continued longhole blasting operations on the BAC drill horizons; • Cave stalls does not propagate as fast as planned will directly impact on rates of draw and hence production; • Increased host rock dilution enters the cave prematurely from highwall failures in the open pit or propagation into the carbonaceous shales, reducing rock value and economic mine life; • Ring fall-off; • Undercut bridging and hence drawbell delays; • Failure to complete the drill and blast operation, leading to more localized or narrow caving; and • Plug collapse as the cave approaches the pit bottom, with implications for pit bottom activities such as dewatering	• Hydrogeologically coupled cave simulations have been prepared using latest site data and available technology to predict the conditions required to cave. As more information becomes available these simulations should be re-run to facilitate changes to mine design, strategy, or schedule; • Mine design includes provision for cave recovery drill drives above the expected caving horizon, from which mass blasts can be drilled to assist in cave propagation; • Hydraulic fracturing could be employed as a pre-conditioning technique further above the cave back to promote cave development; • Draw control plans, an even rate of draw across the footprint, but for limited periods draw can be slowed slightly, to encourage cave growth in slower areas, to combat chimneying; • Comprehensive monitoring will be implemented to track cave progression and provide early warnings of adverse behavior to allow the mine to react with, where required, additional blasting or development to provide drilling vantage; and • The mine will have at its disposal a fleet of drill & blast equipment capable of continuing LHS mining should the cave fail to propagate naturally.
Extraction Level Performance	The small footprint of the extraction level results in a relatively small number of drawbells. Any loss of drawbells or drawpoints will influence targeted production performance, and can materially impact on cave draw and, if unmanaged, dilution entry. Causes for drawpoint loss or underperformance may include:	• The advanced undercut sequence alleviates some of the risks for adverse ground behavior on the extraction level during construction; • Further improvements to definition of ground conditions on the extraction level, with future surface drill programs;

Risk Statement	Description/Cause/Consequence	Mitigation
	• Poor construction execution of drawpoints resulting in unfavorably shaped bells or under supported drawpoints; • Geotechnical instability of primary and secondary apex pillars placed in the ore/waste contact; • Bridging or point loading of pillars with oversize, structurally compromising the drawpoint; and • Excessive oversize and hangup frequencies due to initial coarse caving with limited secondary fragmentation, poor blast performance, or undercutting of or slabbing from longhole rings.	• Geotechnical mapping during development will define specific support requirements for each drawpoint, which may deviate from those specified in GCMP; • Probe drilling will help define contacts and provide opportunity to optimise layouts to avoid the more adverse contacts; • Geotechnical simulations have identified potentially problematic drawpoints which will receive additional intensive support; • Instrumentation is planned to monitor extraction level ground conditions, to provide early warnings of adverse behaviour; • Productivity assessments account for moderate oversize and hangup frequencies, and equipment budgets include provision for drawpoint clearing equipment including mobile breakers and high hangup rigs, to quickly remove oversize material stuck in drawpoints; • Maintain high drilling accuracy to reduce poor blast-induced oversize; and • Design aims to achieve an overall 50-degree blasting face, to reduce undercutting of main rings, which can be further steepened should development identify structural features that may cause failures.
Mine Flooding	Mine flooding to point at which infrastructure is damaged, production is halted, and/or the mine is evacuated for safety reasons will drive varying degrees of cost, revenue loss, and health and safety consequences from minor to catastrophic. Flooding could occur as a result of: • Groundwater and stormwater that exceed estimated volumes; • Sudden inrush of perched water (mudrush); • Extended power failures to dewatering systems; • Mechanical failure of dewatering infrastructure or ruptured dewatering pipes; • Obstructions in dams, pipes, or pumps rendering dewatering system inoperable; and	There are several mitigation efforts underway and/or in place including: • Regular updates to, and calibration of, hydrogeological models to inform dewatering system design; • Inclusion of >50,000 m^3 of flood capacity below the extraction level in design; • Diesel generator back-up power is on site and operational in case of grid failure; • Probe drilling will be conducted during lateral and vertical development through standpipes, and lateral development grouting plans and equipment are in hand; • Establishment of sumps and pumps prior to development in areas with known water intersections;

Risk Statement	Description/Cause/Consequence	Mitigation
	• Faulty design or poor installation of drainage systems, sumps, pumps, or electrical infrastructure. As an example, submerging positive displacement pumps and associated electrical equipment at 285 lvl could perpetuate mine flooding, damage other critical infrastructure, and stop mining for a long period; removing water from inundated areas could be a significant task that drives loss of revenues and major capital cost to reactivate the mine.	• Water-tight doors planned to garner additional flood capacity UG; • Stormwater monitoring, TARPs and flooding evacuation scenarios have informed rescue timelines under varying inflow rates, and should be updated as plans and inputs evolve; • Designs account for strainers in dams and chains in drain holes for clearing obstructions in water paths; • P/S is equipped with dual dewatering columns in Shaft with isolation valves, and all stations are equipped with dual leg pressure reducers with isolation valves; • Continued efforts to identify, model, and grout unabandoned exploration holes; and • Protective bunds will be constructed around the pit crest to reduce catchment area for storm events.
Mudrush	Sudden inrush of water and/or debris to the mine may harm people and equipment, forcing production hiatus, investigations, and costly restart requirements. There are several potential causes for mudrush including: • Cave propagates through aquiclude into groundwater source and/or perched groundwater causing sudden inrush; • Mechanical failure / downtime of material handling system which results in a stoppage of LHD mucking, settling of fines, and build-up of water; • Underestimated volume of clay or clay-like sediments within the cave, which when mixed with water can turn into a liquid or plastic state; and • Stormwater events channel precipitation and sediment through the open pit into the cave.	• TARP systems will appropriately monitor and trigger responses to changing conditions which could result in mudrushes. Reference Karowe Cave Management Plan; • Mobile production equipment fleet is specified with tele remote functionality to allow remote operation in areas of risk; • Extraction level is graded to central sumps with overflows to flood horizons capable of containing large volumes of stormwater and debris; • Design includes temporary ore storage at the extraction level to permit continuous mucking from drawpoints for maintaining movement and mixing of the muck pile during periods of crushing/hoisting downtime; • Pit dewatering equipment of appropriate capacity will prevent formation of pit lakes prior to breakthrough and will continue to operate after breakthrough for as long as safe to do so; • Pit crest bunds will aim to minimize surface runoff into the pit; and • Dewatering wells around the open pit are designed to lower the phreatic surface, with deeper wells planned to target lower lithologies.

Risk Statement	Description/Cause/Consequence	Mitigation
Airblast	Excessive draw, beyond the rate of cave propagation, creates a large air gap. This, combined with sudden release of large blocks, or plug-type failure of the cave back, could generate air blasts that have the potential to injure people and equipment in its path. Areas of potential exposure include the extraction level, and the longhole drilling horizons (whilst D&B operations are continuing). Causes include poor / excessive draw practice; and inadequate cave back and air gap monitoring.	• Mine designs consider the mucking of swell only, to minimise air gap, until caving is established; • A Cave Management Plan provides guidance on a range of conditions that can arise and is the basis for developing a series of SOP's for operations; • The instrumentation and monitoring strategy focuses on tracking cave propagation through smart markers, slough metres, open hole CMS and a microseismic system. Air gap monitoring is an integral part of the strategy; and • Established industry practice for air gap management will be applied.
Corrosion of Underground Equipment and Materials	Unexpected or early-onset corrosion may result in increased costs and schedule disruptions, particularly affecting critical path activities related to shaft steelwork, primary services, and material handling infrastructure. Groundwater tests show high TDS, equivalent to seawater, and will cause rapid corrosion if not managed properly. Potential causes include: • High TDS groundwater inflows or dewatering lines leaking onto shaft infrastructure (i.e., steelwork, piping, electrical system); • Poor drainage or ventilation around fixed infrastructure; • Procurement or fabrication does not follow corrosion management plan; • Damage during installations compromises factory applied corrosion protection; • Underestimated frequency of, or failure to perform, inspections and maintenance of infrastructure required to mitigate corrosion, as well as scaling issues; • Underestimated life expectancy of equipment and infrastructure with corrosion protection measures; and • Underground mining in highly saline conditions is uncommon in the Southern African region and standard practices adopted in other regions such as Australia where highly saline conditions are common, are not well understood.	• Top Hat shaft guides have been selected for the shafts for their ability to prevent trapped water, as opposed to square hollow type, are constructed of thicker material for longer life, and are quicker to replace when required; • Fixed plant corrosion management plan is well established, considers high saline groundwater chemistry, is issued with procurement packages, and includes instructions for touch-ups post-installation; • Critical equipment and infrastructure have been specified with corrosion specific paint systems and upgraded materials of construction on components; • Regular shaft inspections and routine maintenance programs will identify areas of concern; and • Dual dewatering lines allow for one operational and one standby for maintenance purposes during periods of low-flow

Risk Statement	Description/Cause/Consequence	Mitigation
Diamond Recovery	Diamond recovery directly informs revenue. Potential causes of lower than planned diamond recovery include: • Unavoidable diamond breakage through blasting, secondary cave fragmentation, or rehandling to the plant; • Unplanned high dilution from carbon rich sedimentary units and overwhelming XRT sorters; and • The expected rate of Legacy diamond recovery is not achieved, or Legacy diamond pricing does not meet historical values.	• Chutes and bins are designed to reduce height of fall and some with level controls to maintain a material bed for minimizing impact against liners; • Designs leverage where possible conveyor transfer over mobile equipment digging to handle ore; • Blast hole diameters and powder factors are designed with similar specifications to current open pit to achieve similar blast fragmentation; • XRT sorting trials have successfully tested the ability to separate carbonaceous host rock from diamonds. Cave simulations do not predict >1% dilution from carbonaceous host rock; and • Legacy diamond frequency is statistically estimated from many years of operational data in kimberlite domains which continue at depth – whose diamond grade and value is expected to continue.
Inaccuracies associated with Mineral Resource and Reserve estimation	Unexpected variability in shapes, volumes, densities, tonnages, grades and values of dominant resource domains (EM/PK(S) and M/PK(S)) beyond acceptable uncertainty limits for Indicated resource classification could affect Reserve estimates, mining layouts and mine plans. The current resource estimate at UG extraction level of 310 - 410masl contains approximately 80% volume of EM/PK(S). Minor volume units at the same level amount to 4.5% volume. Ongoing investigations may lead to expansion of the minor units to a potential maximum of 15% volume, largely at the expense of the EM/PK(S) unit.	• Additional infill core drilling from surface and from underground development drives to refine external pipe margins and internal domain boundaries in areas of sparse drillhole coverage; • Lateral mine development in and around the pipe will improve delineation of pipe contacts on multiple levels well in advance of stoping and will inform final detailed designs; • Mapping and bulk sampling of UG development drives at the extraction level and drilling horizons will increase confidence in the continuity of geology, grades and values of the dominant domains at depth and will provide information on the minor units that may enable upgrading of their classification from Inferred to Indicated resources; • Sampling of new core for petrography and microdiamond analysis to improve understanding of local variability within resource domains; • Bulk sampling of stockpiles to confirm grades and current diamond prices; and • The minor volume units are classified as Inferred resources and consequently treated as zero grade dilution in the Reserve

Risk Statement	Description/Cause/Consequence	Mitigation
		estimate. This is extremely conservative as available microdiamond information suggest grades in the range 5 - 15 cpht (Table 14-6).
FRD Dam Breach	Dam breach of FRD 3 has shown potential to inundate some parts of the plant, with population at risk and requiring costly repair of infrastructure. Potential causes for FRD 3 breach include: • Seismic activity and liquefaction; • Overtopping; • Seepage at the toe of the facility or rise of the phreatic surface leading to saturation of the Kalahari / aeolian sand foundation; and • Poor quality of construction or QAQC.	• The mine is located in a low seismic area and is at low risk; • The FRD as designed can accommodate a volume well beyond the FS production and consequently has a low risk of overtopping; • Daily, weekly, monthly, quarterly, and annual facility inspection by appointed personnel and performance reports prepared by the Engineer of Record detects early warning signs of adverse conditions; • Filter drains protect the saturation of the Kalahari sands and are regularly monitored; • FRD 3 design includes the provision of a toe wall (one-third the height of the facility, 10 m wide at 1V:3H) to contain high phreatic conditions; and • FRD 3 dam breach assessment included assessment for potential protection berm.

25.3 Opportunities

Several opportunities have been identified during the FS that could improve project economics, reduce risk or improve execution. Table 25-2 highlights some of the more significant opportunities that will be or are being explored.

Table 25-2: Identified Project Opportunities

Opportunity	Explanation
Contained Value of Inferred Resources	The South Lobe contains minor Inferred kimberlite domains of which >1 Mt will be unavoidably extracted as part of the caving process. These domains are treated as zero grade dilution in the Reserve Estimate, however, could positively impact project revenues if diamond revenue proves positive, a highly likely outcome.
Resource expansion	The South Lobe is open at depth. Expansion of Indicated resources below 250 masl could increase mine life and NPV.
Forward Modeling of Legacy Diamond Revenue	Based on production statistics from 2013 to 2024, Lucara has developed a reliable proprietary revenue model for Legacy diamonds (> $5 million). The model permits forecasting of attributable revenue on an annualized basis, thereby improving cashflow projections for the forthcoming UG phase of mining.
Increased production rate after stope blasting is complete in 2029	The shaft skips, feeding conveyors, and underground crusher are designed to manage up to a peak 3.1 Mt/a. Subject to cave propagation rate and drawpoint mucking capacities there is opportunity to increase production from underground once drill and blast campaigns are complete.
Increased Cave Recovery	There are approximately 540 kct diamonds not mobilized by the cave in the South Lobe between 495 masl and 665 masl of which 160 kct (30%) could be reasonably expected to be liberated through recovery blasting with a portion reporting to the draw points. The mine plan has budgeted a limited amount of capital development to support cave recovery blasts which could aid in this effort.
Lower Groundwater Inflow Rate	The bulk of the mine development will be performed in granite where groundwater will be fracture driven with problematic areas likely grouted. As such the demand on dewatering systems may be reduced, especially during initial development periods, for an OPEX reduction for pumping water from UG, water treatment, and disposal.
FRD 3 Cost Optimization	Given the continued revisions to the LOM plan and the reduced storage requirement for FRD 3, there is now an opportunity to optimise the proposed FRD 3 footprint. This optimisation would allow a reduction in both construction and eventual closure costs.
FRD 3 inundation area to the plant reduction	The current FRD 3 tailings dam breach assessment indicates that a portion of the process plant would be inundated in a failure scenario. However, with optimization of the FRD 3 layout, it is likely that only a single paddock will be required. In this context, prioritising the currently proposed location for paddock 3B, being the furthest from the plant—could help reduce potential inundation impacts on the plant in the event of a breach.

26 RECOMMENDATIONS

Based on the results of this study the UGP shows positive economics 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. The UGP EPCM works and existing KDM operations are supported by KDM's multi-disciplinary technical service team, with staff and budget allowances suitable to address many of the recommendations outlined below. Costs outside of the existing budgets as included in this FS are identified with cost estimates.

26.1 Geology, Mineral Resource and Reserve

Recommendations for further geology and resource work focuses on efforts to increase confidence in key areas and continue to advance the understanding of the Mineral Resource:

- Evaluate additional drilling, and underground mapping and geological assessment as UG development drives advance to determine the distribution of the KIMB3 unit and other minor units intersected in drilling below 500 masl, and to better constrain the modelled but not drill-confirmed extent of the M/PK(S) domain below 438 masl elevation. These works are currently outside of the existing planned programs, but valued at approximately US$ 3.0 million;
- Continued incorporation of pit geological mapping to enhance pipe contact and internal kimberlite domain definition until the end of OP life using KDM's geological resources;
- Monitoring 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 using KDM's geological resources;
- Perform discrete microdiamond and mini-bulk sampling of the minor kimberlite units currently classified as Inferred Resources from development ore, using existing site budgets and procedures for discrete production samples from the OP and stockpiles;
- Continued reconciliation of production forecasts relative to mine production; and
- Bulk sampling of stockpiles to confirm grades and current diamond prices.

26.2 Geotechnical

The study has identified and addressed the critical risks. Additional predictive modelling is suggested if significant changes should be detected for the current understanding of the geology and hydrogeology of the pipe. Additional data is recommended to fill data gaps, specifically for the extraction and undercut levels. The following recommendations are made:

- Optimise the planned design and execution of geotechnical instrumentation drilling and installation including Geo4Sight Smart markers, and Ramjack Smart slough metres, and

geophones to monitor cave performance, and expand plans to include shaft infrastructure monitoring;

- Leverage geological drilling discussed above to better define the contacts and rock properties through the extraction and undercut levels;
- Perform more representative contact definition study work, incorporating leached granite domains, with a focus around the extraction level;
- Regularly review and revise the GCMP to capture changes to designs or geotechnical conditions;
- Provide guidance (oversight) and training for on-site geotechnical staff in caving and cave operations;
- Plan and implement and integrated geotechnical system that allows appropriate storage and interpretation of field, lab, modelling, I&M and operational data;
- Develop fit for purpose SOPs using the Cave Management Plan; and
- Draft a geotechnical execution plan (operational readiness).

Most of the work will be performed internally using KDM's technical services and EPCM team, however, additional 3rd party support may be required to interpret results from the drill program and update any necessary structural models. Additional 3rd party support is not expected to cost more than US$0.5 million.

26.3 Hydrogeology

Groundwater monitoring should continue with efforts focused on improving knowledge of the basement granites, contacts, and fracture corridors. The mine should leverage underground probe holes, drain holes, and dewatering infrastructure with instrumentation to refine understanding of hydrogeologic parameters and improve the overall hydrogeological models, inflow rates, and downstream mine and infrastructure designs including:

- Advance the planning for a groundwater monitoring program to target deeper lithologies, contacts between the kimberlite and host rock, leached granite, and fracture corridors;
- Install pressure valves in selected cover holes or drain holes opportunistically to increase the groundwater level monitoring points in the Mea and granite units;
- Install planned flow metres on dewatering lines to quantify the water pumped out of the mine, which will roughly reflect the inflow rates;
- Continue to monitor water chemistry to inform material and equipment specifications, maintenance plans, and replacement schedules and budgets;

- Perform regular update and calibration of groundwater flow model with the measured inflow rates and water levels during the development and mining operation. The model should be updated if the simulated results differ noticeably from the field observations or the geologic model and geologic structure model are significantly modified; and

- Refine the design and requirements for lower drain hole groups in relation to the underground mining schedule and implementation schedule to maximise the inflow rate of lower drain groups.

Most of the work will be performed internally using KDM's technical services and EPCM team including water model updates. However, additional costs for periodic model calibration by 3rd party may be required at additional, unbudgeted cost, not expected to exceed US$0.1 million per update.

26.4 Mining

Mine plans are well defined with shafts physically sunk, lateral development initiated, and design and procurement of major infrastructure including crushing and dewatering systems complete. Forward-looking recommendations focus on filling design gaps in production drilling, blasting, and extraction plans which will be required for engaging with contractors responsible for performing the work and procuring outstanding equipment and infrastructure. Additional opportunities observed through engineering, design, and schedule reviews should be investigated to improve on the UGP.

- Optimise cave draw simulations as additional data becomes available to improve mine recovery, dilution, and initiation points, at an estimated additional cost of US$0.2 million;

- Evaluate secondary egress plans to supplement the planned mine refuge installations. Ladderways would be constructed, connecting sublevels, at an estimated additional cost of US$1.5 million.

- Improve definition of kimberlite contacts and optimise detailed development layouts to minimise any stability impacts of contact zones, to be performed using on site resources and planned development probe drilling;

- Advance the underground installation of cave monitoring systems in the schedule by leveraging development opportunities;

- Refine extraction level design following procurement of production equipment;

- Engage with explosive manufacturers to secure in-country supply and training on the use of wireless detonators, required for stoping methods used before natural caving takes place;

- Investigate cave recovery drill & blast design with review of cave draw simulations and available underground development;

- Commission mechanical evaporators to reconcile assumed system efficiencies, and update designs and schedules for facility expansions;

- Early engagement with critical contractors (production drilling, UG construction, UG electrical) to ensure availability, skillset, and alignment in scope approach;
- Refine plans for OP closure, stormwater bunding, and the management of pit sumps up to crown pillar failure;
- Monitor opportunities to replace production Load Haul Dump (LHD) machines with electric or battery operated once the initial fleet of diesel powered LHDs have reached their useful life span;
- Prepare detailed shaft maintenance and hoisting schedules to confirm available working times and structure workforce;
- Reconcile shaft cycle times, capacities, and availability following successful commissioning in 2026 to inform work schedules, mine plans, and production forecasts;
- Train existing technical services team in mine planning, scheduling, cost and production forecasting including the use of 3D mine, ventilation, geology, and geotechnical software; and
- Continue Operational Readiness efforts to prepare for operation and maintenance of permanent infrastructure and establish operating procedures and response plans for underground activities including TARP advancement of measurable thresholds for reaction during operations.

26.5 Tailings

Based on the data reviewed and conclusions drawn from this study update, the QP recommends the following.

26.5.1 FRDs

- Continue with Current Deposition: Continue with the current deposition plan for FRD 2, incorporating recommended improvements in slurry velocity as per (Knight Piésold, 2024) and improving deposition techniques by depositing along three embankment walls (current deposition is along 2 walls). This will maximise the utilization of FRD 2's capacity, ensuring alignment with operational objectives and efficient use of existing infrastructure;
- Expansion of FRD 2 and introduction of FRD 3: Due to the reduced LOM and storage capacity requirements, challenges in achieving target beach slopes on FRD 2, and increased tailings density, the designed storage capacity of FRD 3 has been affected. To meet the updated 2025 LOM requirements, it is essential to expand FRD 2 beyond its original 10 m height design from 2022 design. Additionally, introducing FRD 3 to provide supplementary capacity is key. This can be achieved by adding one paddock from the current FRD 3's design (this could reduce the inundation area to the plant) or optimizing its overall footprint due to the continued decrease in its capacity requirement; and

- Conduct Regular Capacity Modelling: Establish a regular schedule for capacity and deposition modelling, to be reviewed at least every three years. This will help adapt to changes in production forecasts, material density, and construction planning. This proactive strategy will support effective planning for the FRDs lifecycle construction and deposition, ensuring continued alignment with operational demands.

26.5.2 CRD

- Complete Current Deposition: Continue with the existing plan for Leg 2 of the conveyor system to deposit in a southerly direction until reaching its final position. This ensures alignment with current operational goals and efficient use of existing infrastructure;

- Introduce a Fourth Leg: To meet the revised LOM tonnage requirements, introduce a fourth conveyor leg. This addition addresses the increased production output and the new deposition strategy, which emphasizes capacity expansion and reduced need for rework on existing ramps; and

- Conduct Regular Modelling: Implement a schedule for regular deposition modelling, reviewed at least every three years, to adapt to changes in production forecasts, material density, and moisture content. This proactive approach will help in planning the end-of-life deposition strategy effectively and ensure continued alignment with operational needs.

26.5.3 Environmental

Based on the data reviewed and conclusions drawn from this study update, the QP recommends the following.

- Periodic evaluation of future disposal needs, options, and costs for produced saline groundwater alongside of already recommended periodic review and calibration of the hydrogeological model.

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28 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
BEV	battery electric vehicle
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

Symbol / Abbreviation	Description
CHP	combined heat and power plant
CIM	Canadian institute of Mining and Metallurgy
CK	coherent kimberlite
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cP	centipoise
cpm^3	carats per cubic metre
cpht	carats per hundred tonnes
ct/stn	carats per stone
c/t	carats per tonne
Cr	chromium
CRD	coarse residue deposition
ct	carat or carats
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

Symbol / Abbreviation	Description
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
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 m^2)
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

Symbol / Abbreviation	Description
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
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
lvl	mining level
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

Symbol / Abbreviation	Description
m^2	square metre
m^3	cubic metre
m^3/day	cubic metres per day
m^3/h	cubic metres per hour
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 or mL	metres above mean sea level
MAP	mean annual precipitation
Mb/s	megabytes per second
mbg	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

Symbol / Abbreviation	Description
Nc	critical speed
NG	normal grade
NGL	natural ground level
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
RFD	revenue frequency distribution
RH	Royal Haskoning
RL	Reduced Level.
RMR	rock mass rating
ROM	run of mine

Symbol / Abbreviation	Description
rpm	revolutions per minute
RQD	rock quality designation
RVK	resedimented volcaniclastic kimberlite
s	second (time)
Scfm	standard cubic feet per minute
SABS	South African Bureau of Standards
SANS	South African National Standards
SEDEX	sedimentary exhalative
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

Symbol / Abbreviation	Description
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
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

Rock Type / Unit	Description
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)
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
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

29 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

Matthew Moss, P.Eng.

I, Matthew Moss, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Senior Engineer with JDS Energy & Mining with an office at Suite 900, 999 West Hastings Street, Vancouver British Columbia;
3. I am a graduate of the Montana Technological University, with a Bachelors Degree in Mining Engineering, 2011. I have practiced my profession full time since 2011;
4. I have spent the last 14 years working on mining projects; where I have performed underground mine design, planning, scheduling, cost estimating, and management for mining projects;
5. I am a Registered Professional Engineer in British Columbia (#44729);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
6. I have visited the visited the Karowe Mine regularly since 2020 and most recently on Nov 23, 2024;
7. I am responsible for Sections 1-5, 16 (except 16.3, 16.4), 18 (except 18.4), 20.4, 20.5, 24-27 of this Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have participated in previous Karowe mine evaluations including the last published 2023 Feasibility Study;
10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) “Matthew Moss, P.Eng.”

Matthew Moss, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Brandon Chambers, P.Eng.

I, Brandon Chambers, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Senior Engineer with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;
3. I am a graduate of Queen’s University, with a B.Sc. in Mining Engineering, 2010, and with a MASc. in Mining Engineering, 2012. I have practiced my profession continuously since 2012;

   I have spent the last 13 years working in technical, operations, and management positions at mines in Canada and Southern Africa. I have performed mine designs & plans, infrastructure designs & plans, cost estimation, economic modelling, procurement, and operations & construction management for mining projects;

   I am a Registered Professional Mining Engineer in British Columbia (#45269);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I visited the Karowe Project Site initially in 2019. Between 2022-2024 I made regular visits, approximately 3 visits per year;
5. I am responsible for Sections 12.3, 12.12, 15, and 21-22 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have participated in previous Karowe mine evaluations, including the 2019 Feasibility Study (Doerksen et al., 2019) and 2023 Feasibility Study (Doerksen et al., 2024);
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) “Brandon Chambers”

Brandon Chambers, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Hermanus Grütter, P.Geo., Ph.D

I, Hermanus Grütter, P.Geo., Ph.D, do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently under contract as an Associate Principal Geologist with SRK Consulting (Canada) Inc. with an office at 2600–320 Granville St., Vancouver, BC, V6C 1S9;
3. I am a graduate of the University of Cape Town (South Africa), with a Bachelor of Science (Honours) in Geology obtained in 1986 and the University of Cambridge (United Kingdom) with a Ph.D. degree in Geology obtained in 1993. I have practiced my profession intermittently since 1987;

   I have spent the last 32 years working on pre-resource to feasibility-level minerals exploration or mining projects; where I have performed core logging and sampling, exploration project design and supervision, assessment and interpretation of assay results, resource estimates and project development potential, and analysis of economic models for mining projects;

   I am a Registered Professional Geologist in British Columbia (Licence #28007);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I have visited the Karowe Mine on May 1-3, 2024;
5. I am responsible for Sections 12.2, 14 and 19 of this Technical Report;
6. I am independent of the Issuer and related companies applying all the tests in Section 1.5 of the NI 43-101;
7. My prior involvement with the property that is the subject of this Technical Report pertains to (i) peer review of Karowe Mineral Resource Estimates effective July 1, 2019 and June 30, 2023 and (ii) interim Qualified Person responsible for the Karowe mineral resource during 2024Q2
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) “Hermanus Grütter, P.Geo., Ph.D”

Hermanus Grütter, P.Geo., Ph.D

CERTIFICATE OF QUALIFIED PERSON

Kimberley Webb, P.Geo.

I, Kimberley Webb, P.Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Principal Consultant with SRK Consulting (Canada) Inc. with an office at 320 Granville Street, Suite 2600, Vancouver, BC, V6C 1S9;
3. I am a graduate of Rhodes University (South Africa), having obtained the degrees of B.Sc. (Hons.) in Geology in 1994 and M.Sc. in Geology in 2001. I have practiced my profession continuously since 1996;

   I have been an independent consultant for 19 years, contributing to projects across Canada, in southern, central and west Africa, India, Brazil and Australia. I have developed and audited geological models of kimberlites and lamproites, designed drilling, logging and sampling programs in support of diamond exploration, evaluation, resource estimation and mining, and performed technical report writing;

   I am a Registered Professional Geoscientist in British Columbia, Canada (#151489) and a Registered Professional Natural Scientist (Geological Science) in South Africa (#400053/02);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have visited the Karowe Diamond Mine on Jun 14-22, 2017; Jun 11-15, 2018; May 8-17, 2019; Oct 29-Nov 6, 2024; Apr 22-30, 2025; and Nov 10-20, 2025;
5. I am responsible for Sections 6-11, 12.1, and 23 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
7. My prior involvement with the property that is the subject of this Technical Report includes geological development and modelling work reported in Nowicki et al. (2018) while employed as Principal Geoscientist at Mineral Services Canada Inc. (MSC), and geological and resource development work reported in Doerksen et al., (2019) and Doerksen et al., (2024);
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) “Kimberley Webb, P.Geo.”

Kimberley Webb, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Kelly McLeod, P.Eng.

I, Kelly McLeod, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. am currently employed as a President with K-Met Consultants Inc. with an office at 14650 Oyama Road, Lake Country, British Columbia, V4V 2C7;
3. I am a graduate of McMaster University with a Bachelor of Engineering, 1984. I have practiced my profession intermittently since 1984;

   I have spent the last 20 years working on mining projects, where I have been consulting in the mining industry in metallurgy and process design engineering;

   I am a Registered Professional Engineer in British Columbia (15868);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have not visited the Karowe Mine;
5. I am responsible for Sections 12.4, and 13 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have participated in previous Karowe mine evaluations including the last published 2023 Feasibility Study;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 28, 2026

(Original signed and sealed) “Kelly McLeod, P.Eng.”

Kelly McLeod, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Thagarajh Rangasamy, MSc, FSANIRE, MSAIMM

I, Thagarajh (aka Trevor) Rangasamy, MSc, FSANIRE, MSAIMM, do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently the Managing Director, Owner, and Senior Principal Rock Engineer at Middindi Consulting Pty Ltd, Unit 43 & 44, Block C, Willowbrook Office Park, 15 Van Hoof St, Ruimsig, Roodepoort, 1724, South Africa;
3. I hold a Bachelor of Science in Applied Geology awarded by the University of Natal, South Africa (1991), a Master’s in Engineering (Mining) awarded by the University of the Witwatersrand, Johannesburg, South Africa (2004), and a Post Graduate Diploma in Rock Engineering awarded by the University of the Witwatersrand, Johannesburg, South Africa (1999);

   I have been involved in the South African mining industry for 34 years in various roles including research and development (CSIR), production rock engineering (AngloGold-Ashanti, Placer Dome, Barrick Gold and South Deep), project development and implementation and consulting (Itasca, Groundwork, Middindi). I consider myself by reason of my education, my affiliations, and my experience to be a Qualified and a Competent Person as defined in Regulation 14.1.8 of the MHSC (RSA), Act No.29 of 1996;

   I am a Fellow in good standing of the South African Institute of Rock Engineering (FSANIRE, F101075), a Member in good standing of the Southern African Institute of Mining and Metallurgy (MSAIMM, M702075), and a Member of the International Society of Rock Mechanics (ISRM, M101075);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have visited the Karowe Mine on Feb 24 - 28, 2025;
5. I am responsible for Sections 12.5 and 16.3 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
8. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) “Thagarajh Rangasamy, MSc, FSANIRE, MSAIMM”

Thagarajh Rangasamy, MSc, FSANIRE, MSAIMM

CERTIFICATE OF QUALIFIED PERSON

Houmao Liu, Ph.D., PE

I, Houmao Liu, Ph.D., PE, do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Principal Hydrogeologist with ITASCA Denver with an office at 143 Union Blvd., Suite 525, Lakewood, CO 80228, USA;
3. I am a graduate of the University of Colorado, with a Ph.D., 1992. I have practiced my profession intermittently since 1992;

   I have spent the last 30+ years working on mining projects; where I have performed, mining hydrogeology, mine dewatering and depressurisation design, numerical groundwater flow, solute transport, and water quality modeling, project management, and supervision of mining hydrogeology for mining projects;

   I am a Registered Professional Engineer in Colorado;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have visited the Karowe Mine on April 26 to 28, 2022;
5. I am responsible for Sections 12.6 and 16.4 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have provided hydrogeologic review to the property that is the subject of this Technical Report, including participation in the 2023 Feasibility Study (Doerksen et. al, 2024);
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 23, 2026

(Original signed and sealed) “Houmao Liu, Ph.D., PE”

Houmao Liu, Ph.D., PE

CERTIFICATE OF QUALIFIED PERSON

Mehrdad Nazari, MSc, MBA, MIMMM

I, Mehrdad Nazari, MSc, MBA, MIMMM, do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am the proprietor and managing director of Prizma LLC, duly registered at 208 Tuzigoot Lane, Wausau, WI 54403, USA, which provides ESIA (Environmental and Social Impact Assessment), ESG (Environment, Social and Governance), IESC (Independent Environment and Social Consultant), IAM (Independent Accountability Mechanism), and licensed Assurance or Verification services;
3. I received my BSc (equivalent) in Mineralogy at the J.W. Goethe University, Germany, in 1987, an MSc in Hydrogeology from the University of Birmingham, UK, in 1991, and an MBA from Henley Business School, UK;

   Over the past 30 years, I have worked as (i) a Hydrogeologist, and an Environment, Health and Safety audit Consultant in the German offices of an international environmental and engineering firm, (ii) serves as a Principal Environmental Specialist at a multilateral financial institution based in the UK, focusing on extractive projects, (iii) headed Corporate Responsibility/ESG Research at an ESG rating agency based in London, UK, before doing business as an independent consultant as Prizma LLC for approximately 20 years, conducting due diligence reviews, impact assessments, sustainability reporting, providing verifications or assurance services mainly involving mining projects or operations based in Botswana, Brazil, Burkina Faso, Chile, China, Ecuador, Guatemala, Indonesia, Ivory Coast, Kyrgyzstan, Mexico, Mongolia, the Pacific Ocean (deep-sea mining), USA, and Venezuela;

   I am a Member of the International Association for Impact Assessment and (IAIA, based in the USA), and the Institute of Material, Mineral & Mining (IOM3, based in the UK), and an Accountability licensed assurance service provider;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

   I have been supported in my work for this Technical Report by Dr. Donald S. Proebstel, a Senior Associate with Prizma LLC with over 30 years of international experience, including as VP Environmental Affairs & Sustainability at Gold Reserve Inc.; Senior E&S Analyst with a development finance institution; Senior Analyst Pike Research (now Navigant Research); and Senior Environmental Analyst at AATA International. Dr. Proebstel’s experience in the mining sector includes Equator Principles reviews of projects in Argentina and Chile, reviewing and troubleshooting an ESIA for a brownfield underground gold mining projects in Guatemala; reviewing and monitoring the environmental performance of a gold mining operation in Kyrgyzstan; conducting GHG and climate-related analysis for gold and copper mining projects and operations in the Ivory Coast and the USA, conducting MAC TSM verifications for a diamond mine in Botswana, and contributing to an international arbitration related to a copper-gold mining project in Venezuela; Dr. Proebstel was awarded a B.S. Fishery Biology and a PhD in Conservation Biology from the Colorado State University. He is also Md a member of the IAIA;
4. Our recent site visits to Lucara’s Karowe Mine in Botswana include the following: for Mehrdad Nazari: 13-17 February 2023, 14-25 February 2022, and 27-28 April 2021, and for Dr. Proebstel 21-25 November 2024, 14-25 February 2022, and 27-28 April 2021;
5. I am responsible for Sections 12.8 and 20 (except 20.4, 20.5) of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. Dr. Don Proebstel and my previous involvement with Lucara and the Karowe Mine includes contributing or generating sustainability reports, biodiversity, social impacts or management plans, and human rights reviews, as well as providing ESG-related training, assurance and verification services, such as MAC TSM;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 26, 2026

(Original signed and sealed) "Mehrdad Nazari"

Mehrdad Nazari, MBA, MSc, MIMMM

CERTIFICATE OF QUALIFIED PERSON

Justin Michael Teixeira, Pr. Eng.

I, Justin Michael Teixeira, Pr. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Principal Civil Engineer with Knight Piésold (Pty) Ltd with an office at 1 Discovery Place, The Ridge, Sandhurst, Sandton, Gauteng 2196, South Africa;
3. I am a graduate of the University of Pretoria, with a bachelor’s degree in civil engineering, 2012. I have practiced my profession continuously since 2012;

   I have spent the last 13 years working on mining projects; where I have performed tailings designs and management. I have extensive knowledge on the day-to-day operations of tailings storage facilities, with a focus on quality control and risk management. I have designed tailings facilities from concept to detailed design and has overseen projects through all the phases of development from start up to completion. I also perform the Engineer of Record (EoR) duties for several of these facilities;

   I am a Registered Professional Engineer with Engineering Council of South Africa (20190614) and the Engineers Registration Board of Botswana (201904347);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have visited the Karowe Diamond Mine regularly since 2019 and most recently on Nov 25, 2025;
5. I am responsible for Sections 12.11 and 18.4 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have participated in previous Karowe Diamond Mine evaluations including the last published 2023 Feasibility Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 27, 2026

(Original signed and sealed) “Justin Michael Teixeira, Pr. Eng.”

Justin Michael Teixeira, Pr. Eng.

CERTIFICATE OF QUALIFIED PERSON

Lehman Myburgh van Niekerk, Pr. Eng.

I, Lehman Myburgh van Niekerk, Pr. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Diamond Mine 2025 Feasibility Study Technical Report” (the “Technical Report”) with an effective date of September 30, 2025 prepared for Lucara Diamond Corp;
2. I am currently employed as a Principal Process Engineer with DRA Global; with an office at Building 33, Woodlands Office Park, 20 Woodlands Drive, Woodlands, Sandton, 2080;
3. I am a graduate of the Northwest University (South Africa), with a B. Eng Chemical Engineering (with specializing in minerals processing) degree, 2002. I have practiced my profession intermittently since 2003;

   I have spent the last 23 years working on mining projects; where I have performed production engineering & process design, study management, proposals engineering, cost estimation and project controls, and commissioning management for mining projects;

   I am a Registered Professional Engineer (registration number: 20070182) as well as a member of the Southern African Institute of Mining and Metallurgy (SAIMM, registration number: 704697) in South Africa;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;
4. I have visited the Karowe Mine on Sep 2-3, 2019, and Oct 14-16, 2025;
5. I am responsible for Sections 12.9, and 17 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had prior involvement with the property that is the subject of this Technical Report. The past involvement is related to the Phase 1 and Phase 2 Karowe Diamond Mine Plant design and commissioning aspects, as well as the 2023 Feasibility Study (Doerksen et al., 2024);
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 30, 2025
Signed Date: January 24, 2026

(Original signed and sealed) “Lehman Myburgh van Niekerk, Pr. Eng.”

Lehman Myburgh van Niekerk, Pr. Eng.