KAROWE MINE UNDERGROUND FEASIBILITY STUDY TECHNICAL REPORT

BOTSWANA

Prepared by:
JDS ENERGY & MINING INC.
Suite 900, 999 W Hastings St.
Vancouver, BC, Canada

Prepared for:
LUCARA DIAMOND CORP.
2000 - 885 W. Georgia Street
Vancouver, BC, Canada

Qualified Persons	Company
Gord Doerksen, P.Eng.	JDS Energy & Mining Inc.
Trace Arlaud, Reg. Mem. SME	JDS Energy & Mining Inc.
Kelly McLeod, P.Eng.	JDS Energy & Mining Inc.
Carly Church, P.Eng.	JDS Energy & Mining Inc.
John Armstrong, Ph.D., P.Geo	Lucara Diamond Corp.
Andrew Copeland, Pr.Eng	Knight Piésold
Johan Oberholzer, Pr.Eng.	Royal Haskoning
Matthew Pierce, P.Eng.	Pierce Engineering
Markus Reichardt, Ph.D.	Reichardt & Reichardt
Kimberley Webb, P.Geo	SRK
Cliff Revering, P.Eng.	SRK
Koos Vivier, Pri.Sci.Nat	Exigo
Lehman van Niekerk, Pr.Eng.	DRA Projects

Effective Date: September 26, 2019
Report Date: December 16, 2019

Date and Signature Page

This report entitled Karowe Mine Underground Feasibility Study Technical Report, effective as of September 26, 2019 was prepared and signed by the following Qualified Persons (QPs):

Original document signed and sealed by:

Gord Doerksen, P.Eng. *December 16, 2019*
Gord Doerksen, P.Eng. Date Signed

Original document signed and sealed by:

Trace Arlaud, Reg. Mem. SME *December 17, 2019*
Trace Arlaud, Reg. Mem. SME Date Signed

Original document signed and sealed by:

Kelly McLeod, P.Eng. *December 16, 2019*
Kelly McLeod, P.Eng. Date Signed

Original document signed and sealed by:

Carly Church, P.Eng. *December 16, 2019*
Carly Church, P.Eng. Date Signed

Original document signed and sealed by:

John Armstrong, P.Geo. *December 14, 2019*
John Armstrong, P.Geo. Date Signed

Original document signed and sealed by:

Andrew Copeland, Pr.Eng. *December 15, 2019*
Andrew Copeland, Pr.Eng. Date Signed

Original document signed and sealed by:

Johan Oberholzer, Pr.Eng. *December 14, 2019*
Johan Oberholzer, Pr.Eng. Date Signed

Original document signed and sealed by:

Matthew Pierce, P.Eng. — *December 13, 2019*
Matthew Pierce, P.Eng. — Date Signed

Original document signed and sealed by:

Markus Reichardt, Ph.D. — *December 14, 2019*
Markus Reichardt, Ph.D. — Date Signed

Original document signed and sealed by:

Cliff Revering, P.Eng. — *December 16, 2019*
Cliff Revering, P.Eng. — Date Signed

Original document signed and sealed by:

Kimberley Webb, P.Geo. — *December 13, 2019*
Kimberley Webb, P.Geo. — Date Signed

Original document signed and sealed by:

Koos Vivier, Pr.Sci.Nat. — *December 13, 2019*
Koos Vivier, Pr.Sci.Nat. — Date Signed

Original document signed and sealed by:

Lehman van Niekerk, Pr.Eng. — *December 17, 2019*
Lehman van Niekerk, Pr.Eng. — Date Signed

PARTNERS IN
ACHIEVING
MAXIMUM
RESOURCE
DEVELOPMENT
VALUE

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Gordon Doerksen, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Mine Underground Feasibility Study Technical Report, Botswana” with an effective date of September 26, 2019, (the “Technical Report”) prepared for Lucara Diamond Corp.;

2. I am currently employed as President – Engineering 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 Montana Tech with a B.Sc. in Mining Engineering, 1991.

   I have worked in technical, operations and management positions at underground and open pit mines in Canada, the United States and Zambia without interruption from 1985 to 2006. I have worked continuously as a mining consultant from 2006 to present and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and project management for dozens of mining projects worldwide including co-authoring numerous 43-101 Technical Reports;

4. I am a Registered Professional Mining Engineer in British Columbia (License No. 32273);

5. 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 visited the Karowe Mine site on; April 18, 2018, December 12-13, 2018, February 18-27, 2019, March 20-27, 2019, April 25-27, 2019, May 14-15, 2019, June 5-11, 2019 and July 22-24, 2019;

7. I am responsible for the Executive Summary and Sections 1-5, 12, 13.1, 13.2, 13.4, 15, 16.6.1, 20.5, 23, 24, 26-29 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 provided high level consulting work for the Karowe Mine prior to conducting this FS. My past work included a review of open pit mining contractor performance and underground mining method options.

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 26, 2019
Signing Date: December 16, 2019

[original signed and sealed] "Gordon Doerksen, P.Eng."

GORDON DOERKSEN, P.Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Tracey Arlaud, Registered Member of the SME., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Project Director 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 the La Trobe University, Melbourne, Australia with Bachelor Science with Honours in Geology (B.Sc. Hons.) 1996 (complete the course 1994); University of Ballarat, (Now Federation University), Ballarat, Victoria Australia with Graduate Diploma of Mining 2001 (completed the course 2000), Masters of Mining Engineering (M.Eng.) 2007 (completed the course 2006), from the University of Ballarat (Now Federation University), Ballarat, Victoria Australia. I have practiced my profession continuously since 1994;

   I have worked in technical, operations and management positions at mines in Australia and Indonesia. I have been a consultant for over fifteen years and have performed mine design, mine planning, detail design, cost estimation, operations, project management & construction management, technical due diligence reviews and technical report writing for mining projects worldwide;

   I am a Registered Member of Society for Mining, Metallurgy and Exploration # 4119811

   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 Mine site: May 23rd, 2018 and December 11-13th, 2018.

5. I am responsible for the following sections of this Technical Report; 16 (except 16.3, 16.4, 16.6.1), 21.3.2, 22.2.2.

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. In 2018. I participated in a high level review of a previous ongoing study and a high-level conceptual mining method 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: December 17. 2019
Signing Date: December 17, 2019

(Original signed and sealed) "Tracey Arlaud"

Tracey Arlaud REG MEMBER SME4119811

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as an 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 Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 13 years consulting in the mining industry in metallurgy and process design engineering;

4. 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;

5. I have not visited the Karowe Mine site;

6. I am responsible for 13.3 of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have had no past involvement with the property that is the subject of this Technical Report;

9. 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

10. 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 26, 2019
Signing Date: December 16, 2019

[original signed and sealed] "Kelly McLeod"

Kelly McLeod, P. Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Carly Church, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as an 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 the University of British Columbia, with a B.A.Sc. in Mechanical Engineering, 2006. I have practiced my profession intermittently since 2006.

   I have spent the last 6 years working on mining projects; where I have performed, project engineering & infrastructure design, project management, purchasing and expediting, cost estimation and project controls, economic modelling, construction planning and management for mining projects.

   I am a Registered Professional Engineer in British Columbia (#46451) and the Yukon (#2749);

   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 Mine site on the following dates;
   - April 25 - 27, 2019
   - August 28 - September 5, 2019;

5. I am responsible for 18 (except 18.4 and 18.8), 21 (except 21.3), 22 (except 22.2), 25 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 no past involvement with the property that is the subject of this Technical 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 26, 2019
Signing Date: December 16, 2019

(Original signed and sealed) "Carly Church P. Eng."

Carly Church P. Eng.

CERTIFICATE OF AUTHOR

I, John P. Armstrong, Ph.D. P.Geol., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Vice President, Technical Services with Lucara Diamond Corp with an office at Suite 2000-885 West Georgia Street, Vancouver, BC, Canada V6

3. I graduated from the University of Western Ontario in 1989 (H.BSc.Geology), and the University of Western Ontario in 1997 (Doctor of Philosophy (Ph.D.)), and have practiced my profession continuously since graduation;

4. I have worked in government, exploration, technical, operations and management positions at mines and projects in Canada, and Botswana. I have been involved with mining, production, and diamond sales activities at the Karowe Diamond Mine continuously since October 2013 as an employee of Lucara Diamond Corp.

5. I am a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License # 1697).

6. 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.

7. I am not independent of Lucara Diamond Corporation due to my position as an Officer of the Corporation, as defined in section 1.5 of NI 43-101;

8. I have visited the Karowe Diamond Mine and Lucara Botswana Sales offices on a regular basis since October 2013 with the most recent visit being December 2019 ;

9. I am responsible for Sections 6, 8, 9, 10.1, 10.2, 11, 19 of this Technical Report

10. I am not independent of Lucara Diamond Corporation due to my position as an Officer of the Corporation, as defined in section 1.5 of NI 43-101;

11. I have visited the Karowe Diamond Mine and Lucara Botswana Sales offices on a regular basis since October 2013 with the most recent visit being December 2019;

12. 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

13. 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 26, 2019
Signing Date: December 14, 2019

(Original signed and sealed) "John P. Armstrong, Ph.D. P.Geol."

John P. Armstrong Ph.D. P.Geol.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Andrew Copeland, Pr. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Technical Director with Knight Piésold Consulting. with an office at 4 De La Rey Road, Rivonia, Johannesburg South Africa, 2128.

3. I am a graduate of the University of Cape Town with a B.Sc. in Civil Engineering, 1987. I have practiced my profession continuously since 1988;

   I have worked in technical, operations and management positions at mines in South Africa, Botswana, Namibia, Zimbabwe, Zambia, DRC, Mali, Ghana, Tanzania, Peru, Chile, Brazil, Venezuela, Canada, the United States and Australia. I have spent 20 years at Anglo American, 1 year at Gold Fields and been a consultant for over six years at Knight Piésold Consulting. I have performed tailings management designs including all related water studies, cost estimation, operations & construction management, audits and reviews of designs and operational facilities, technical due diligence reviews and technical report writing for mining projects worldwide;

   I am a Registered Professional Civil Engineer in South Africa (940040);

   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. On my behalf Justin Teixeira (12th December 2018 and 2nd & 3rd September 2019), Mlungisi Motsa (17th July and 1st & 2nd August 2019) and Keneth Matotoka (26-28th June 2019) visited the Karowe Mine site to carry out geotechnical investigations, site inspections, meet with mine personnel;

5. I am responsible for Section 18.8 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 no past involvement with the property that is the subject of this Technical 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 26, 2019
Signing Date: 15 December 2019

(Original signed and sealed)

Andrew Copeland, Pr. Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, G J Oberholzer, Pr. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Project Manager with Royal HaskoningDHV with an office at 21 Woodlands drive, Woodmead, Johannesburg, Gauteng, 2191, South Africa;

3. I am a graduate of the University of Pretoria with a B.Sc. in Electrical Engineering, 1977. I have practiced my profession continuously since 1977;

   I have worked in technical, operations and management positions at mines in South Africa. I have been an independent consultant for over 30 years and have performed mine engineering designs, cost estimation, technical due diligence reviews and technical report writing for mining projects in Africa;

   I am a Registered Professional Electrical Engineer in South Africa (#810126);

   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 Mine site 9th to 11th October 2017, 4th to 5th July 2018, 6th to 8th November 2018;

5. I am responsible for Sections 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 had no past involvement with the property that is the subject of this Technical 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 26, 2019
Signing Date: 14 December 2019

(Original signed and sealed) " G J Oberholzer Pr. Eng."

GJ Oberholzer, Pr. Eng.

PIERCE
ENGINEERING
THE SUPPORTING MECHANISM

1501 West 28th St.
Minneapolis, Minnesota 55408
(612) 201-7560
matt@pierce-engineering.com

CERTIFICATE OF AUTHOR

I, Dr. Matthew Pierce, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Karowe Mine Underground Feasibility Study Technical Report, Botswana” with an effective date of September 26, 2019, (the “Technical Report”) prepared for Lucara Diamond Corp.;

2. I am currently self-employed doing business as Pierce Engineering LLC with an office at 1501 West 28th St., Minneapolis, MN, USA, 55408;

3. I received my B.S. in Geological Engineering and M.S. in Mining Engineering from Queen’s University in Canada in 1995 and 1997 respectively, and my Ph.D. in Mining Engineering from the University of Queensland in Australia in 2010. I have practiced my profession continuously since 1995, providing consulting geomechanical engineering services to underground and open pit mines with a focus on rock mass characterization, geomechanical mine design analysis and third-party review. I offer expertise in the estimation of rock mass properties and the analysis of caving and collapse potential, fragmentation, subsidence, ore recovery and infrastructure stability.

   I am a Registered Professional Engineer in the Province of Ontario, Canada;

   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 Mine site Dec 11-13, 2018 and Feb 21-27, 2019;

5. I am responsible for Section 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. I have had no past involvement with the property that is the subject of this Technical 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 26, 2019
Signing Date: December 13, 2019

(Original signed and sealed)

Dr. Matthew Pierce, P. Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Markus Reichardt, Ph.D. (Restoration Ecology), do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently Managing Partner of Reichardt & Reichardt, with offices at 78 Buckingham Ave Craighall Park, Johannesburg, South Africa, and 100 Waranderglaan Sterrebek, Belgium;

3. I am a graduate of Queen's University, Canada with a MA (History) 1989 and a Ph.D. in Restoration Ecology from the University of the Witwatersrand, South Africa, 2013.

   I have worked in a variety operational and corporate line functions for companies operating in the diamond, coal, gold and base metal sector across Sub-Saharan Africa, with the last corporate role being Corporate Environmental Manager for AngloGold Ltd. (now AngloGoldAshanti) with a global brief. I have been an independent consultant for 17 years and have performed technical due diligence reviews, EIA/EMP project management, mine closure liability estimates and specialist advisory commissions for mining projects in over 20 jurisdictions" I have also performed extensive project and company sustainability risk evaluations for asset and investment managers in South Africa and Europe.;

   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 Mine site on September 9-11, 2017, October 14-18, 2018 and December 3-6, 2018:

5. I am responsible for Section 20 (except 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. I have had no past involvement with the property that is the subject of this Technical 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 26, 2019
Signing Date: December 14, 2019

[original signed and sealed] "Markus Reichardt, Ph.D."

Markus Reichardt, Ph.D.

SRK Consulting (Canada) Inc.
2200–1066 West Hastings Street
Vancouver, BC V6E 3X2

T: +1.604.681.4196
F: +1.604.687.5532
vancouver@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Kimberley Webb, P. Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Principal Consultant with SRK Consulting (Canada) Inc. with an office at Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia, V6E 3X2;

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 over 12 years, contributing to projects across Canada, in southern, central and west Africa, India and Australia. I have developed and audited geological models of kimberlites and lamproites, designed drilling, logging and sampling programs in support of exploration, evaluation, resource estimation and mining, and performed technical report writing.

   I am a Registered Professional Geoscientist in British Columbia, Canada (License # 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 visited the Karowe Mine site during June 11-15, 2018 and May 8-17, 2019;

5. I am responsible for Sections 7 and 10.3 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. My prior involvement with the Karowe Mine 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);

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 26, 2019
Signing Date: December 13, 2019

(Original signed and sealed) "Kimberley Webb, P. Geo."

Kimberley Webb, P. Geo.

SRK Consulting (Canada) Inc
205–2100 Airport Drive
Saskatoon, SK S7L 6M6

T: +1.306.955.4778
F: +1.306.955.4750
saskatoon@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Cliff Revering, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;
2. I am currently employed as Principal Consultant with SRK Consulting (Canada) Inc. with an office at Suite 600, 350 3rd Ave. North, S7K 6G7, Saskatoon, Saskatchewan, Canada.
3. I am a graduate of the University of Saskatchewan in 1995 with a B.E. in Geological Engineering and completed a Citation in Applied Geostatistics from the University of Alberta. I have practiced my profession continuously since 1995. My relevant experience includes more than 24 years employment in the mining industry, related to exploration, mine operations and project evaluations, with a specialization in geological modelling, mineral resource and reserve estimation, production reconciliation, grade control, exploration and production geology and mine planning.
4. I am a professional Engineer registered with the Association of Professional Engineers and Geoscientists of Saskatchewan (APEGS#9764).
5. 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 visited the Karowe Mine site May 14-17, 2019;
7. I am responsible for Sections 1.7, 12.2 and 14 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 had no past involvement with the property that is the subject of this Technical Report;
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.

Saskatoon, Saskatchewan
December 16, 2019

Cliff Revering, P.Eng, CPAG, BE.
Principal Consultant (Geological Engineering)

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Jacobus Vivier (Hydrogeologist, Pr.Sci.Nat)., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of December 12, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Project Director with Exigo Sustainability (Pty, Ltd) with an office at 70 Regency Street, Route21 Business Park, Centurion, Pretoria, South Africa.

3. I am a graduate of the Institute for Groundwater Studies at the University of the Free State with an M.Sc. in Hydrogeology, 1996 and a PhD. in 2011 from the North-West University in South Africa. I have practiced my profession continuously since 1996;

   I have worked in technical feasibility projects, for mines in South Africa, Botswana, Mozambique, DRC, Saudi Arabia and mining operations in South Africa. I have been an independent consultant for over 20 years and have performed practical mine dewatering, dewatering strategy and designs, mine water and waste management, numerical flow and mass transport modelling, water supply options and analysis, hydrogeochemistry cost estimation, technical due diligence reviews and technical report writing for mining projects.

   I am a Registered Professional Hydrogeologist Pr.Sci.Nat no 400177/05.

   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 Mine site;
   a. February 20-22, 2018
   b. May 23-26, 2018
   c. May 30-31, 2018
   d. August 13-14, 2018
   e. September 25-26, 2018
   f. November 12-13, 2018

g. December 3-6, 2018
h. December 12-13, 2018
i. February 20-22, 2019
j. June 4-6, 2019
k. June 18-27, 2019
l. October 31 - November 5, 2019

5. I am responsible for Sections 16.4 and 17.4.9, 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 no past involvement with the property that is the subject of this Technical 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: December 12, 2019
Signing Date: December 13, 2019

(Original signed and sealed) Jacobus Vivier (P.hD., M.Sc Hydrogeology, Pr.Sci.Nat).

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Lehman van Niekerk, Pr. Eng, do hereby certify that:

1. This certificate applies to the Technical Report entitled "Karowe Mine Underground Feasibility Study Technical Report, Botswana" with an effective date of September 26, 2019, (the "Technical Report") prepared for Lucara Diamond Corp.;

2. I am currently employed as Senior Process Engineer with DRA Mineral Projects. with an office at No.3 Inyanga Close, Sunninghill, Johannesburg, Gauteng, South Africa, 2157;

3. I am a graduate of the North-West University, North-West Province, South Africa with a B.Eng. in Chemical Engineering specializing in Minerals Processing, 2003. I have practiced my profession continuously since 2003;

   I have practiced my profession continuously since graduation in 2003 and have operational and project experience in the mineral processing of diamond bearing material;

   I am a I am a registered Professional Engineer with the "Engineering Council of South Africa" (ECSA, No. 20070182) and member of the "Southern African Institute of Mining and Metallurgy" (SAIMM, No. 704697);

   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 Mine site on September 02 and 03, 2019;

5. I am responsible for Sections 17.1, 17.2. 17.3, 17.4.1, 17.4.2, 17.4.3, 17.4.4, 17.4.5, 17.4.6, 17.4.7, 17.4.8 and 17.4.10 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 no past involvement with the property that is the subject of this Technical Report during;

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 26, 2019
Signing Date: December 17, 2019

(Original signed and sealed) "Lehman van Niekerk, Pr. Eng."

____

Lehman van Niekerk, Pr. Eng.

Senior Process Engineer

DRA Projects (Pty) Ltd

NOTICE

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

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

Table of Contents

List of Figures and Tables

1 Executive Summary

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Lucara Diamond Corp. (Lucara) to carry out a Feasibility Study (FS) on extending the life of the Karowe Diamond Mine (KDM) by mining underground (UG) after the completion of open pit mining. This study describes the combined life of mine (LOM) open pit and underground plan as well as highlights the contribution of the UG to the overall plan economics. This report was prepared using the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101.

JDS was assisted in the FS work by specialist consultants including but not limited to:

- Pierce Engineering: Geotechnical Analysis, and Recommendations;
- Itasca: Geotechnical Modelling;
- Exigo Sustainability Pty. Ltd. (Exigo): Hydrogeological Data Collection and Analysis, Mine Dewatering, Water Modelling and Water Management;
- Royal Haskoning (RH): Power Supply;
- Reichardt & Reichardt: Environment and Social;
- DRA: Mineral Processing Description;
- SRK Consulting (Canada, Inc.): Geotechnical Data Collection, Geology, Mineral Resource Estimation and UG Material Flow Simulation; and
- Knight Piésold (KP): Waste Management.

1.2 Project Description

The KDM is an existing open pit mine located in Central Botswana. The mine began commercial operations in July 2012 and currently operates a 2.6 Mt/a processing facility. The KDM has mined and processed approximately 20 Mt of ore since the start of operations. The mine has established itself as one the world's most prolific producers of large, gem quality Type IIa diamonds. Since 2015, KDM has produced two diamonds greater than 1,000 carats in weight, and two of the world's most valuable rough diamonds: the 1,109 carat (ct) Lesedi La Rona (US$53 M) and the 813 carat Constellation diamond (US$63 M).

The in-situ open pit reserve is expected to be fully depleted by 2025. The mine currently has approximately two years of stockpiled kimberlite ore. This FS evaluates extending the mine life by mining underground after completion of open pit mining with processing of stockpiles taking place opportunistically through the mine life. Stockpiles are also expected to provide all of the mill feed material for the last two years of planned mill operations.

The FS UG Project is summarized below:

- Mining:

- 7,200 t/d shaft (approximately 750 m deep) operation utilizing long hole shrinkage mining (2.6 Mt/a);
- 33.5 Mt of UG ore mined at a grade of 15.1 carats per hundred tonnes (cpht) at an estimated value of US$725/carat; and
- Extraction of approximately 400 vertical metres of the AK6 deposit South Lobe from 310 metres above sea level (masl) (700 m below surface) to the bottom of the depleted open pit (approximately 700 masl or 300 m below surface).

- Processing plant throughput of 2.7 Mt/a;
- 5.05 million carats recovered (UG only);
- Five-year UG construction period beginning 2020 and ending in 2025 (to align with the depletion of the open pit); and
- 13 years of UG operations.

The most significant infrastructure upgrade required to support the UG operations will be a new Botswana Power Company (BPC) electricity supply line and a substation at the BPC tie in. The current main substation and distribution systems on the mine site will be expanded. Additional infrastructure upgrades proposed and estimated in this FS include:

- Expansion of the coarse and fine residue facilities;
- Sediment pond and water management structures;
- Construction camp;
- Expansion of the existing water and sewage plants and pipelines;
- Additional offices, warehouse, shop, meeting and training rooms;
- Change house, lamp room, lineout room and first aid office; and
- Security facilities.

1.3 Location, Access and Ownership

The Karowe Mine spans approximately 1,523 ha in the Central District of Botswana, 23 km west of the Letlhakane diamond mine and 25 km south of the Orapa diamond mine.

The geographic coordinates of the Karowe Mine are 25° 28’ 13” E / 21° 30’ 35” S.

The mine is accessed via a well maintained, 15 km all-weather gravel road from the paved Letlhakane to Orapa road. Letlhakane is the closest village and can be accessed from the major cities of Gaborone and Francistown by paved roads. The closest airport that is serviced by commercial flights is in Francistown, approximately 200 km away or a two-and-a-half-hour drive. There is also an airstrip within the nearby Debswana controlled Orapa Township. The Karowe Mine site has its own 1,500 m gravel airstrip.

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

1.4 History, Exploration and Drilling

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

Table 1-1: Historical Exploration Programs

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

Source: Lucara (2019)

1.5 Geology and Mineralization

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

The OKF lies on the northern edge of the Central Kalahari Karoo Basin along which 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 Karowe is sub-outcropping flood basalt of the

Stormberg Lava Group (~130 m thick), underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup (~345 m thick), below which is the granitic basement.

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

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

The geological model presented in this report (Figure 1-1) is updated from that presented in the previous Technical Report (Nowicki et al., 2018). Modifications include revisions to the pipe margin to reflect recent mining gains in all three lobes, and changes to the pipe shell and internal domain model of the South Lobe based on recent core drilling. The most significant changes are extension of the base of the model by 190 m (from 256 to 66 masl), reduction in the volume of M/PK(S) below 500 masl, and modelling of an additional internal domain encompassing the areas where drilling to date indicates KIMB3 is most prevalent. The pipe shells of the Centre and North Lobes have also been updated based on the recent core drilling.

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

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

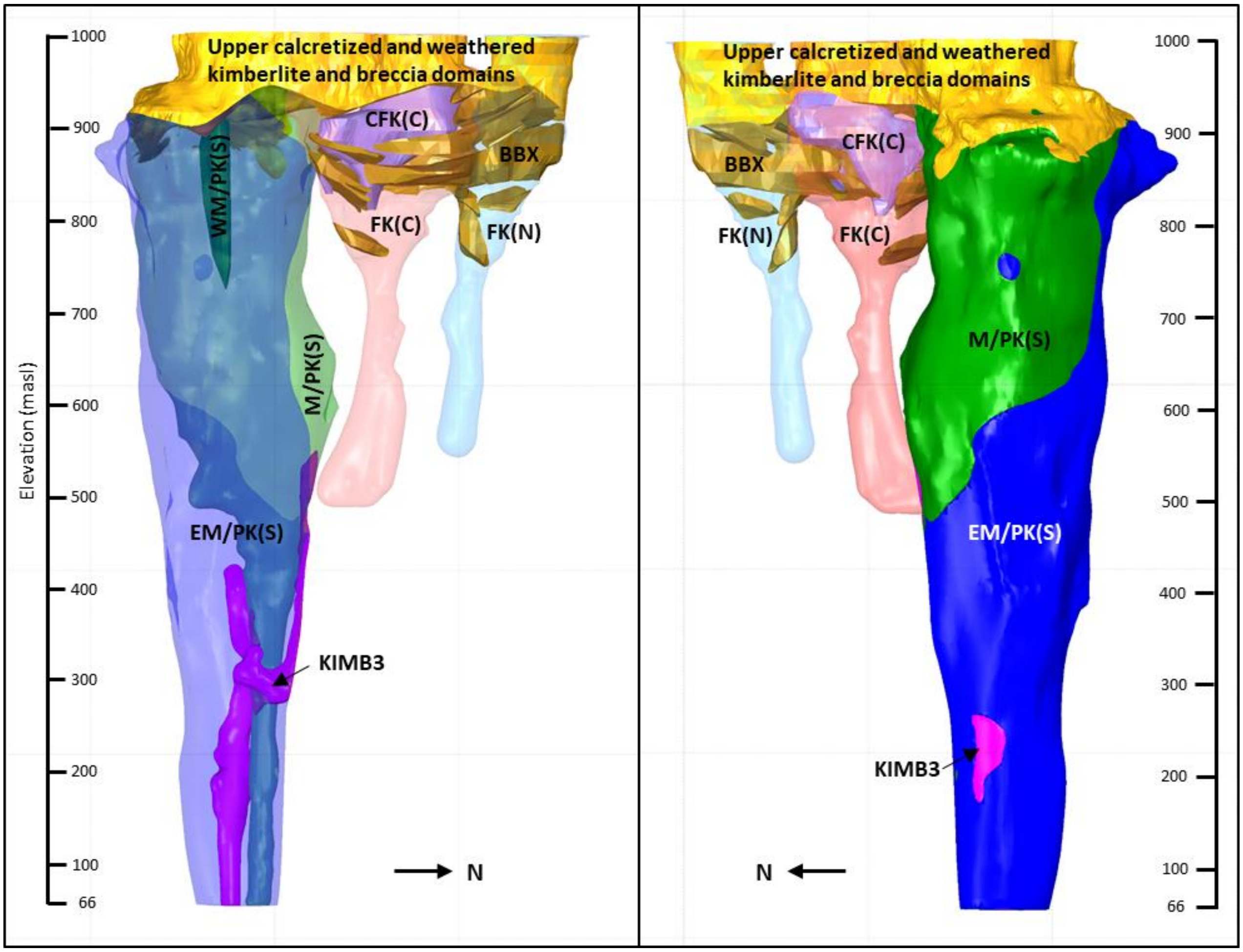
Lobe	Unit	Domain	Description
North	BBX	BBX(N)	Country rock breccia
	CKIMB	CKIMB(N)	Calcretized kimberlite
	FK(N)	FK(N)	Fragmental kimberlite
	KBBX	KBBX(N)	Kimberlite and country rock breccia
	WBBX	WBBX(N)	Weathered country rock breccia
	WK	WK(N)	Weathered kimberlite
Centre	BBX	BBX(C)	Country rock breccia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

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

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

Source: SRK (2019)

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



Source: SRK (2019)

1.6 Mineral Processing Test Work

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

1.6.1 Comminution Test Work

Comminution test work to determine the characteristics of the deeper kimberlite ore was conducted at Base Metallurgical Laboratories (BaseMet) in Kamloops, Canada. Bulk samples and HQ drill core representing EM/PK(S) and M/PK(S) zones of the South Lobe were taken at various depths through the deposit. Bulk samples were taken from the current open pit at approximately 900 masl. Diamond drill core was sampled from varying depths below the open pit and within the planned UG mining zone. The test work was carried out 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 impact on production rate when deeper UG material is processed.

The comminution test work completed on the bulk samples included: crushing work index, Bond Rod and Ball Mill work indices, and JK drop weight. The drill core test work included Bond Rod and Ball Mill work indices and SMC.

The results indicate that there is not a significant difference in hardness between the EM/PK(S) and M/PK(S) material. The samples tested demonstrated similar hardness characteristics to the material presently being processed in the AG Mill, and therefore, the planned UG ore can be processed in the current comminution circuit without a loss in throughput.

1.6.2 XRT Test Work

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

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

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

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

1.7 Mineral Resource Estimate

The 2019 Mineral Resource update for the Karowe Diamond Mine incorporates historical drilling and sampling data obtained prior to 2018, and additional drilling and sampling information obtained in 2018 / 2019 which targeted the delineation of the deep extension of South Lobe (deeper than approximately 600 m from surface). The 2019 update also includes geological information and production data derived from open pit mining to the end of June 2019. Historic and current geological data was used to develop an updated internal geology model for the South Lobe, and updates to the external contacts for the South, Centre and North Lobes.

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

Value distribution models and estimates of average price per carat (US$/ct) for each kimberlite domain and lobe have been developed from discrete mine production obtained since the start of mining in in July 2012 and reflect the latest diamond sales data to the end of August 2019. The value models exclude all revenue generated from diamonds sold for more than US$10 M each since 2014, which includes the Constellation diamond (813 ct sold for US$63 M) and the Lesedi la Rona diamond (1,109 ct sold for US$53 M).

The 2019 mineral resources for Karowe, as summarized in Table 1-3, have been classified as either Indicated or Inferred Mineral Resources, according to 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 July 1, 2019.

Table 1-3: Karowe 2019 Mineral Resource Statement (effective date of July 1, 2019)

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m^3)	Carats (Mcts)	Grade (cpht)	Average (US$/ct)
Indicated	South_M/PK(S)	9.40	27.81	2.96	3.01	10.8	$631
	South_EM/PK(S)	7.62	22.10	2.90	4.68	21.2	$777
	Centre	1.28	3.28	2.57	0.50	15.1	$367
	North	0.44	1.08	2.45	0.13	11.8	$222
TOTAL INDICATED		**18.74**	**54.27**	**2.90**	**8.32**	**15.3**	**$690**
Inferred	South_M/PK(S)	0.10	0.31	3.05	0.03	10.5	$631
	South_EM/PK(S)	1.40	4.18	2.97	0.87	20.9	$777
	South_KIMB3	0.32	0.94	2.94	0.10	10.9	$631
TOTAL INFERRED		**1.82**	**5.42**	**2.97**	**1.01**	**18.6**	**$750**

Notes:

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All numbers have been rounded to reflect accuracy of the estimate.
2. Mineral Resources are in-situ Mineral Resources and are inclusive of in-situ Mineral Reserves.
3. Mineral Resources are exclusive of all mine stockpile material.
4. Mineral Resources are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond losses within the smaller sieve classes expected within the current configuration of the Karowe process plant.
5. Inferred Mineral Resources are estimated on the basis of limited geological evidence and sampling, sufficient to imply but not verify geological grade and continuity. They have a lower level of confidence than that applied to an Indicated Mineral Resource and cannot be directly converted into a Mineral Reserve.
6. Average diamond value estimates are based on 2019 diamond sales data provided by Lucara Diamond Corp.
7. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery.

Source: SRK (2019)

1.8 Mineral Reserve Estimate

A mine plan has been developed to extract the economic portions of Indicated Mineral Resources of the Karowe Project. The South Lobe is planned to be mined through a combination of open pit and underground mining methods. The North and Centre Lobes are planned for extraction by open pit mining methods only. Open pit designs were prepared by Lucara and the associated mineral reserves were verified by JDS. Underground design, schedule, and reserves estimates were prepared by JDS. A consolidated summary of the Mineral Reserve Estimate, by mining method and pipe, is presented in Table 1-4. Ore stockpiles are included in the Mineral Reserve Estimate.

The effective date for the Mineral Reserve Estimate contained in this report is September 26, 2019 and was prepared by Qualified Person (QP) Gord Doerksen, P.Eng. All Mineral Reserves in Table 1-4 are classified as Probable Mineral Reserves. The Mineral Reserves, except stockpiles, are not in addition to the Mineral Resources, but are a subset thereof.

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

Table 1-4: Karowe Mine Mineral Reserve Estimate

Lobe -Type	Classification	Ore (Mt)	Diluted Grade (cpht)	Contained Carats ('000s ct)	Price (US$/ct)
Open Pit					
North	Probable	0.6	10.0	56	222
Centre	Probable	3.2	15.1	478	349
South – EM/PK(S)	Probable	3.6	23.9	850	777
South – M/PK(S)	Probable	10.2	10.8	1,098	631
Open Pit	**Total**	**17.4**	**14.2**	**2,481**	**618**
Underground					
South – EM/PK(S)	Probable	16.3	19.9	3,246	777
South – M/PK(S)	Probable	17.1	10.6	1,807	631
Underground	**Total**	**33.5**	**15.1**	**5,053**	**725**
Stockpiles					
North	Probable	0.4	12.7	51	222
Centre	Probable	0.4	12.8	54	349
South – M/PK(S)	Probable	1.6	9.5	151	631
Mixed	Probable	4.0	5.0	198	609
Stockpiles	**Total**	**6.4**	**7.1**	**454**	**542**
Combined					
All	**Total**	**57.3**	**13.9**	**7,988**	**681**

1. Prepared by Gord Doerksen, P.Eng. JDS Energy & Mining Inc.
2. CIM definitions were followed for Mineral Reserves and the effective date of the Mineral Reserve is September 26, 2019.
3. Mineral Reserves are estimated based on an UG mining cost of US$9/t, a processing cost of US$16/t and a G&A cost of US$6/t. Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the mineral resource block model assumptions and therefore have taken recoveries into account. All of the kimberlite material in the South Lobe is above the cut-off value.
4. Diamond valuation was derived from historical sales adjusted for current and estimated future values.
5. Tonnages are rounded to the nearest 100,000 tonnes; diamond grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained diamonds are reported as thousands of carats.

Source: JDS (2019)

1.9 Geotechnical and Hydrogeological Context

An exhaustive geotechnical and hydrogeological data collection program was undertaken in preparation for the FS. The following programs / test work was undertaken:

- Over 21 km of core was logged from geotechnical drill holes (including hyperspectral logging) along with wireline logging (including acoustic televiewer);
- 7,385 field strength tests and over 3,500 laboratory tests encompassing shear strength, uniaxial and triaxial compressive strength, weathering susceptibility and tensile strength;
- Pumping tests from 23 water holes;
- 58 packer tests; and
- 400 hydrogeochemical tests and analyses.

The homogenous nature of the rock units at Karowe has resulted in geotechnical domains that closely follow lithology, with some additional subdomains (e.g. contact zones) established on the basis of weathering. The unweathered granite basement host and South Lobe kimberlite ore are both of very good quality, exhibiting high mean intact strength (UCS=137-146 MPa) and sparse jointing (>10 m spacing). The unusually high strength (and low weathering susceptibility) of the kimberlite eliminates natural caving as an option but presents a good opportunity for stoping. Kimberlite intact strengths are lower where the kimberlite is in contact with the country rock.

The bulk of the host rock above the granite, comprising approximately 345 m of sedimentary rock (shales, mudstones and sandstones of the Karoo Supergroup) and approximately 130 m of igneous rock (basalts of the Stormberg Lava Group) are of good quality, exhibiting intact strengths that are approximately half that of the granite and kimberlite (mean UCS=53-83 MPa) and similar sparse jointing (>10 m spacing).

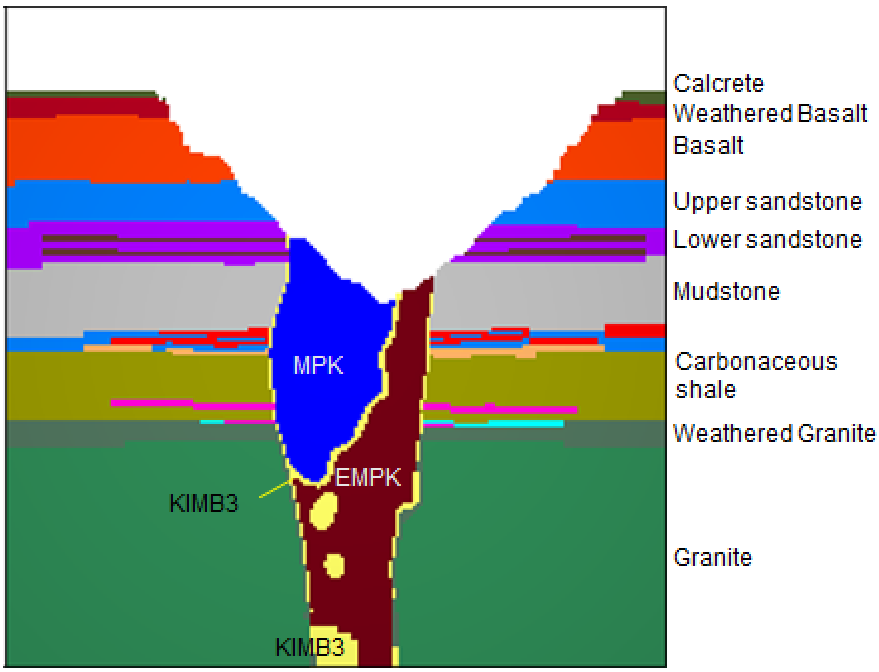
There are some weaker layers within the country rock that exhibit low intact strengths (mean UCS=28-40 MPa). These include the upper Ntane sandstones, the red mudstone beds within the lower Mosolotsane sandstone, some layers within the Tlapana mudstones and the weathered granite. These last two units also have more tightly spaced joints (~1.2-4.4 m spacing, predominantly subhorizontal) than the remainder of the rock on site.

Rock mass classification indicates that the formations in the area of interest have fair to good rock mass quality. The average Laubscher RMR rating is between 50 and 60. The Q' of all lithologies except Kalahari ranges between 200 and 800, which is classified as extremely good to exceptionally good. The RQD for all the formations was 90% and above.

Regional in-situ horizontal stresses are low in the country rock (roughly half of the vertical stress) while the pipe has elevated horizontal stresses, as evidenced by the results of wireline overcoring tests conducted as part of the geotechnical data collection program. There are no major faults evident in the kimberlite or host sediments.

The favorable geotechnical properties of the ore (and much of the host rock) combined with the stable cylindrical shape of the pipe are expected to result in good geomechanical performance, with benchmarking and numerical modelling suggesting limited vertical (ore) and lateral (waste) overbreak (including limited subsidence beyond the final pit crest), high recovery, stable infrastructure and low risk of mud rush, air blast and seismicity. The potential for leaving a competent kimberlite skin against the weaker layers presents a low risk for country rock overbreak and associated lateral dilution.

Figure 1-2: Karowe Lithologies Section View



Source: Itasca (2019)

The water control and hydrogeological context of the deposit and host rocks are key elements in mine planning. The AK6 deposit sits within known, layered, sedimentary, regional aquifers that have been identified since the 1980's and challenges associated with dewatering and depressurization of these aquifers have been experienced by other local mines.

The main water-bearing lithologies are the upper sandstone / basalt contact and the lower sandstone base contact. A fracture zone aligned in a north-north-west strike and at a dip of ±85º to the west is made up of discrete, widely spaced sub-vertical joints that intersect the water-bearing zones and provide a conduit for lateral and vertical water movement. In general, the AK6 kimberlites are not permeable with the exception of the North Lobe contact zone.

The water bearing zones are interbedded with impermeable aquitards made up of grey and red mudstones within the lower sandstone lithology. These aquitards have a persistent head and greatly inhibit the ability to dewater and depressurize both the bottom of the open pit and the proposed underground mine. The red mudstone layer at Karowe is significantly thinner that that seen in nearby operations making it easier to manage both hydrogeologically and geotechnically.

An underground dewatering gallery and drill array are planned to be installed as a priority in the UG mine development and will be developed at the 680 L (about 330 mbs) off of the ventilation shaft during sinking.

The array of UG dewatering holes gives practical dewatering and depressurization control and flexibility that cannot be obtained from surface wells.

The open pit is currently being dewatered using approximately 20 surface wells at a combined pumping rate of 350 m^3/h. This rate must be maintained, at a minimum, to affect the required drawdown of water to the base of the upper sandstone. Below the base of the upper sandstone, dewatering becomes significantly more challenging, resulting in the plan to use an UG dewatering system.

Deeper in the deposit, below the carbonaceous shales, are weathered and solid granites. These can potentially contain localized hot, saline water that will be initially grouted and then drained at a rate of 30-50 m^3/h. Elsewhere in the region, hot saline water is also experienced in the Mea Arkose zone which lies on top of the granite. This unit is not present as a continuous layer at Karowe and has not shown to be water-bearing.

A grout curtain has been planned around the shaft locations to mitigate the impact of the water-bearing zones on shaft development.

1.10 Mining

The Karowe Mine is an existing open pit operation, which has been in production since 2012. Conventional open pit drill and blast mining with diesel excavators and trucks provide an average annual 2.6 Mt of kimberlite feed to the mill. All open pit mining activities are performed by Botswanan mine contractors working 365 days per year on three, eight-hour shifts in the pit and two, 12-hour shifts in the processing facility. The open pit mine operation is expected to terminate mid-2025, ending at an elevation of approximately 700 masl.

There are substantial resources remaining below the economic extents of the open pit that may be extracted by underground mine methods. A 7,200 t/d shaft operation utilizing long hole shrinkage mining (a form of fully-assisted caving) is proposed to provide an additional 13 years of mine life to the Karowe operation after a five-year construction period commencing in 2020.

The Karowe resource contains three distinct coalescing pipes, referred to as the North, Centre, and South Lobe. 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 mbs (from 1,010 masl to 250 masl). The North and Centre Lobes extend below the open pit limit but have been excluded from the planned underground mine as they are inferred at depth and are of low value.

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

Several UG mining methods were investigated as part of this study including block caving (BC), block caving with pre-conditioning, sub-level caving (SLC), and long hole shrinkage (LHS). The small hydraulic radius at depth (27 m), low in-situ (horizontal) stress in combination with high compressive and tensile strength of the kimberlite suggests that the resource will not cave naturally or with pre-conditioning and will therefore require drill and blast assistance. The resource economically favours long hole shrinkage over sub-level caving for its bottom up approach, which takes advantage of the denser and much higher value kimberlite at depth coupled with low operating costs and less development risk.

The LHS method is planned to systematically drill and blast the entire lobe on a vertical retreat basis. The method can be thought of conceptually as a fully assisted cave. In LHS, the blasted muck is left in the excavation during stoping to stabilize the host rock with only the swell extracted / pulled during the drill and blast phase. Mucking takes place from draw points at the bottom of the mine on the 310 Level (L) (310 masl). As ore is blasted, it swells beyond its in-situ volume, and this volume is mucked / pulled from the draw points to maintain a blasting void within the excavation. Once the ore is fully blasted to the bottom of the open pit, the South Lobe is drawn empty by mucking the draw points. There are several advantages to the selected mining method in comparison to an SLC operation, including:

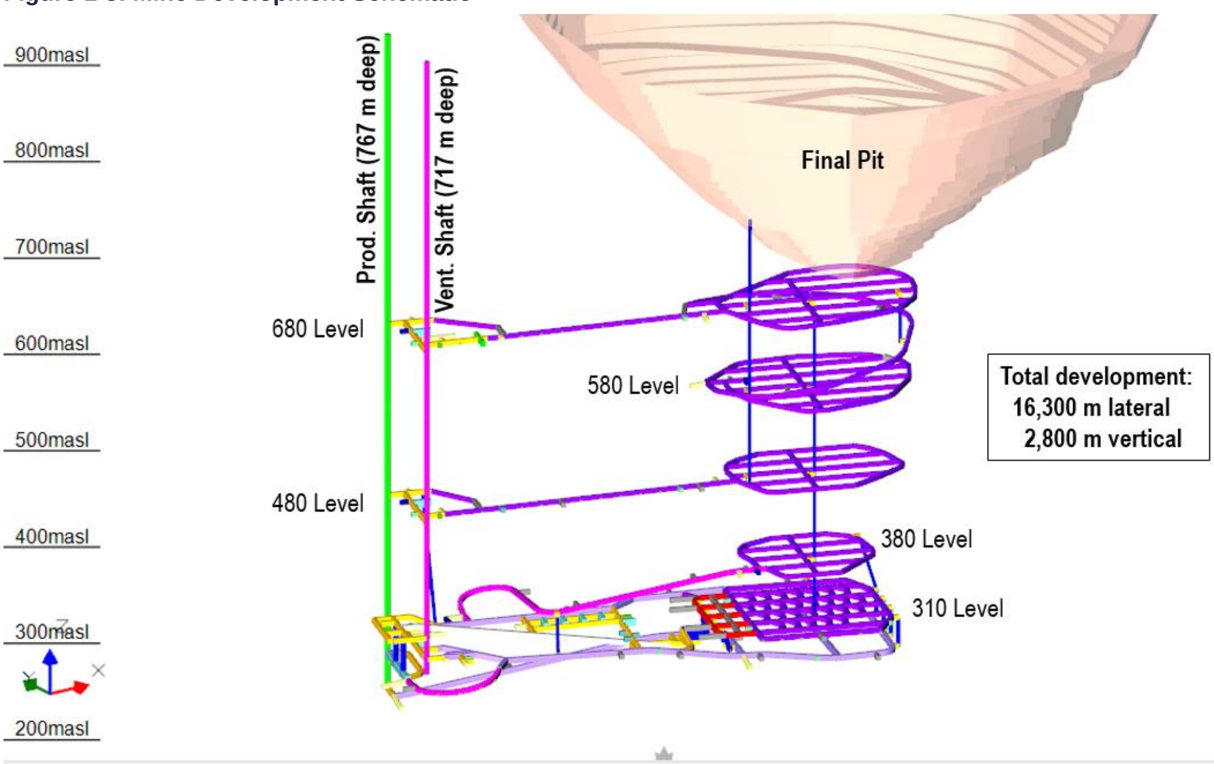
- Mining the highest value first (adds +US$150 M/y in early revenue);
- Much lower and delayed dilution (5% versus +20% for SLC);
- Development and production of the underground can occur simultaneously with pit operations (eliminating reliance on stockpiled OP ore);
- Significantly lower operating costs (less than 50% of SLC OPEX);
- Reduced dewatering risk by using a grouted shaft and delaying surface breakthrough for five production years;
- Reduced ground control risk with minimal development in poor ground (shaft access vs ramp access);
- Significantly reduced metres of development (particularly in poor ground);
- Reduced development and operating labour;
- Extraction level is designed to manage natural caving should it occur;
- Ability to rapidly increase draw once the resource is fully blasted; and
- Ability to economically mine below the 310 L.

Access to the underground mine will be from a 767 m deep production shaft, 7.5 m in diameter, sunk from surface to 245 masl. The shaft will be equipped with two 21-t skips for production hoisting and a service cage for man and material movement through the mine. This shaft will also serve as the main fresh air intake to the mine. A second shaft, 6.0 m in diameter, 717 m deep, driven from surface to 295 masl, will be equipped with a heavy lift hoist for moving large equipment throughout the mine and hoisting development waste during pre-production. This shaft will serve as the main exhaust route and secondary egress for the mine. The two shafts are offset from the kimberlite pipe approximately 375 m northwest of the South Lobe, well outside of the potential subsidence zone, and 100 m from each other. Shafts will be driven blind using conventional drill and blast equipment and will be developed concurrently. Average sinking rates range from 1.2 m/d during the production shaft pre-sink up to 2.5 m/d in the smaller vent shaft through good ground. It is expected to take approximately three years to fully sink and equip both shafts, plus another two years to complete all underground development, capital installations, and production ramp up.

There will be a total of eight working levels in the mine, six of which will be accessed by a shaft station. Levels are named by their elevation in masl. The 310 L will serve as the primary working level and provide access to the main underground infrastructure including production draw points, crusher, and maintenance facilities. Above this level will be four drilling horizons: 380 L, 480 L, 580 L, and 680 L; where production equipment will work to drill and blast stopes. The 380 L will be accessed by ramp from the 310 L. The 480 L and 680 L will be accessed by a dedicated shaft station. The 580 L will be accessed by ramping down from the 680 L through the kimberlite to avoid development in the less competent carbonaceous shale hosted between 520 masl and 650 masl. Near the main 310 L will be the conveyor station at 335 masl, shaft load out station at 285 masl, and the production shaft bottom at 245 masl.

Shaft stations will be developed by the shaft crews and include a primary drive between the two shafts to establish a ventilation connection, as well as sufficient auxiliary drives to install power, water, and air services to support lateral development with conventional rubber tired, diesel mining equipment. Figure 1-3 shows an isometric view of mine development.

Figure 1-3: Mine Development Schematic



Source: JDS (2019)

The underground lateral development will be driven by three development jumbos, initially mobilized to the 310 L. Each crew will drive an average of 3.5 m/d in a priority heading and 2.5 m/d in a secondary heading, to a maximum of 11 m/d per working jumbo. After the majority of the development is complete on the 310 L, one jumbo will be sent up to the 480 L and another up to the 680 L. The last jumbo will remain on the 310 L for any rehabilitation work that will need to be completed throughout the mine life. During pre-production, a total of 15 km of development will be driven.

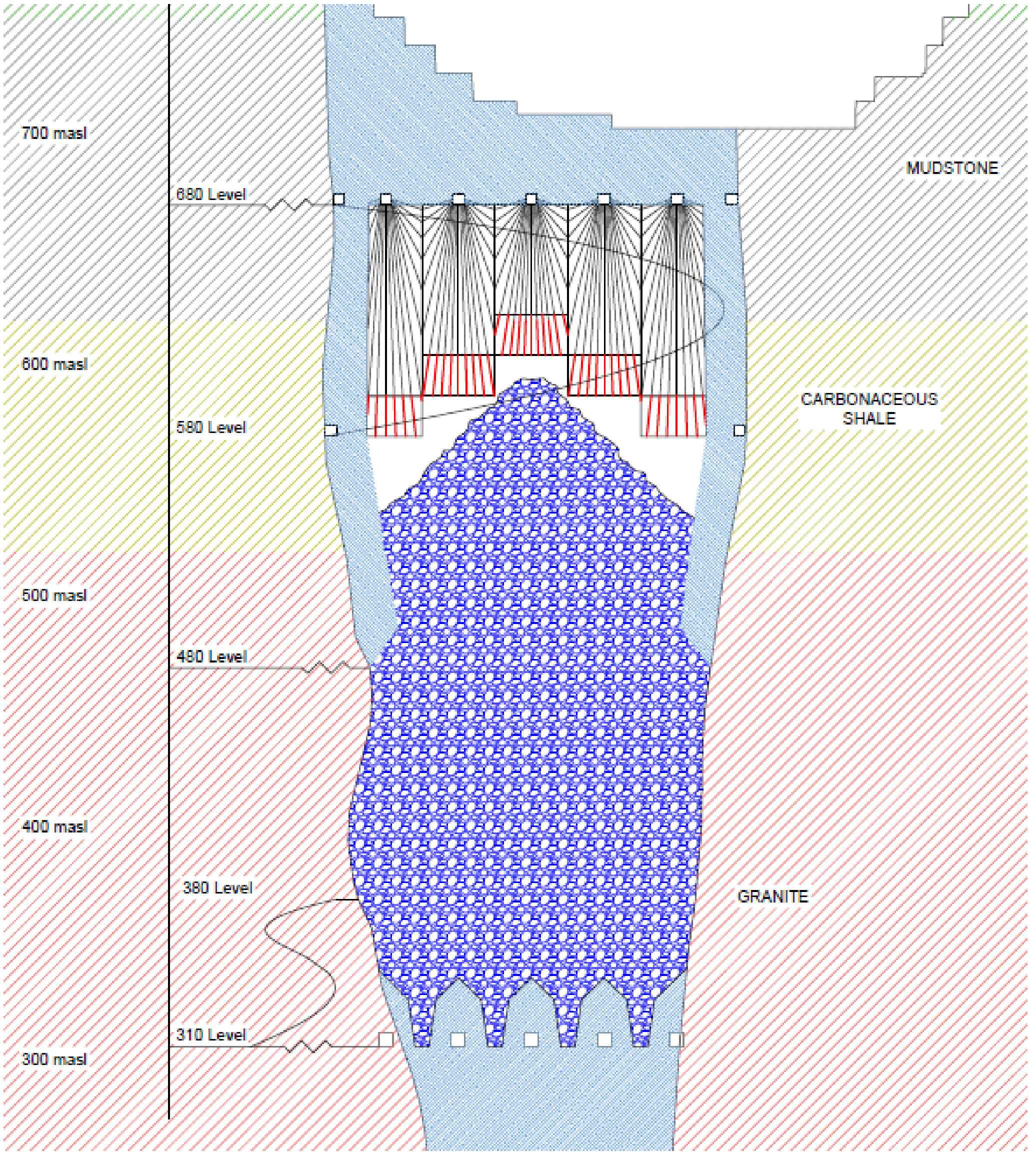
Drill horizons are spaced at 100 m vertical intervals to accommodate the in the hole hammer (ITH) drill's effective drill length of a 150 mm diameter hole. Drilling of the stopes will be completed by mainly down holes on a 4.35 m burden by 5.00 m spacing ring pattern. The average length of hole per ring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

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

Through areas of weaker host rock above the granite, a 15 m skin of kimberlite will be left temporarily around the walls of the lobe to prevent dilution and unraveling. This skin will be recovered later through drilling and blasting during final draw down of the muck pile.

Figure 1-4 illustrates a schematic cross section of the pipe, showing the pyramidal advance of stopes while leaving a 15 m skin of kimberlite along the walls.

Figure 1-4: Mining Method Illustration

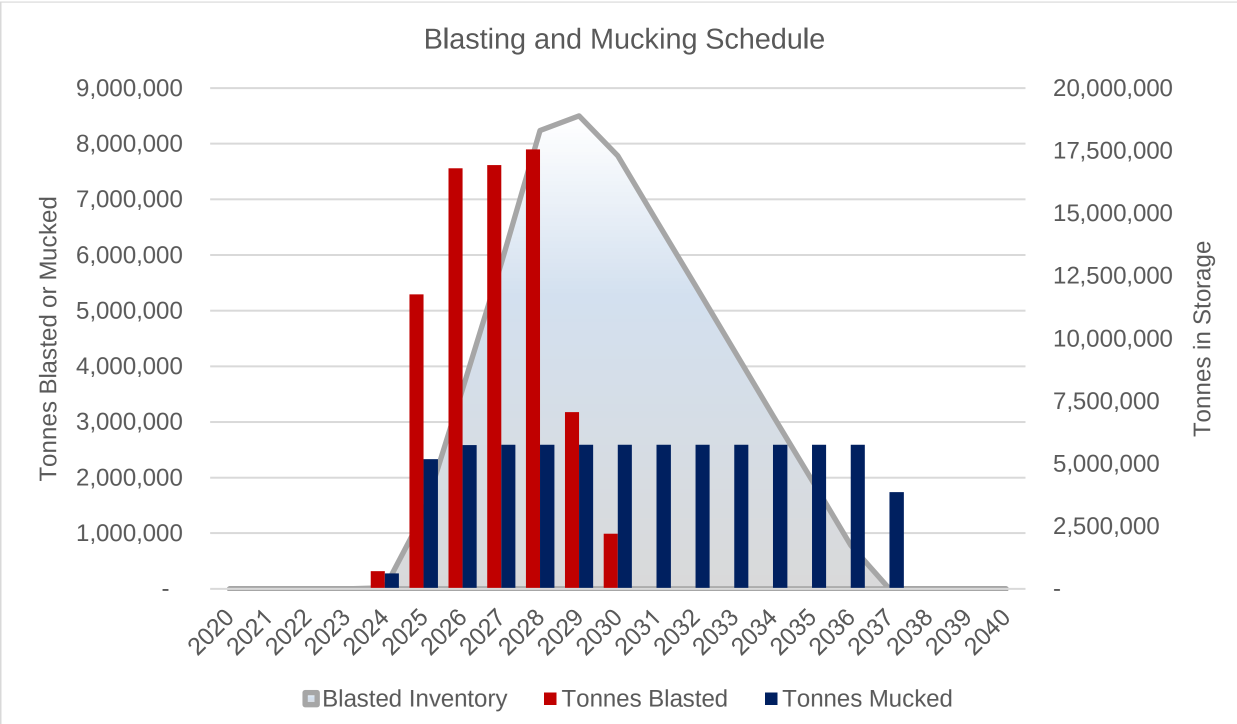


Source: JDS (2019)

Five ITH drills will be utilized to drill and blast approximately 21,000 t/d in order to supply 7,200 t/d of swell to the draw bells for the first six years of operations. Peak broken inventory occurs in year five for a total of 18.9 Mt. After six years, the South Lobe will be fully blasted, and mucking will continue at a constant rate of 7,200 t/d until the underground reserves are depleted at the end of year thirteen. It is important to note that the combination of the kimberlite skin and mining the first half of the stope (200 vertical metres) in granite host rock keeps dilution to a minimum during the first years of underground mining.

The underground blasting and mucking schedule is outlined in Figure 1-5.

Figure 1-5: Blasting and Mucking Schedule



Source: JDS (2019)

The extraction level will be made up of five panels that are driven 31.5 m apart and run the entire length of the lobe. Each panel will access one of 54 draw points driven 18 m x 12 m in a herringbone pattern. The extraction level will contain one perimeter drive to allow traffic to go around panels in the event of a blockage or maintenance at the draw points. At the northwest side of the extraction level, the five panels will access a 1,000 mm static grizzly from three sides. Re-muck bays will be located near the grizzly to allow for continued mucking during crusher maintenance periods and a quick re-handle once the crusher returns to normal operation. There will be approximately 34,000 t of muck storage capacity on the extraction level, equal to 4.7 days of production, and another 66,000 tonnes of available storage elsewhere in the mine. Three 21-t loaders will be required to maintain production at the draw bells. In addition, two 17-t development loaders will be made available to assist with mucking during periods of re-handle or increased haul distances due to panel rehabilitation.

Material dumped onto the grizzly will feed a 1.3 m x 1.5 m (50” x 60”) underground jaw crusher with 960 t/h capacity located 26 m below the extraction level. The jaw crusher discharge conveyor will feed material onto the skip feed conveyor for transport to the 335 L shaft station. The skip feed conveyor will discharge onto a reversible transfer conveyor which will deposit into one of two crushed ore storage bins, each with a capacity of 3,500 t.

The storage bins will discharge onto a skip loadout conveyor which will direct material to one of two 21-t skips. Skips will cycle to surface every two minutes and dump into an elevated bin for direct truck loading.

55-t trucks will load at the shaft and tram ore to the plant or waste to the waste rock storage facility, some two km away.

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

Table 1-5: Underground Production Schedule

Year	EM/PK(S)			M/PK(S)			Total		
	Tonnes	Grade	Carats	Tonnes	Grade	Carats	Tonnes	Grade	Carats
	Mt	cpht	kc	Mt	cpht	kc	Mt	cpht	kc
2023	0.2	18.4	39	0.1	11.1	6	0.3	16.9	45
2024	0.4	18.2	67	0.1	10.3	10	0.5	16.6	77
2025	2.3	19.1	440	0.2	10.4	19	2.5	18.4	459
2026	2.2	19.8	443	0.4	10.6	38	2.6	18.5	481
2027	2.0	20.4	413	0.6	10.8	62	2.6	18.3	475
2028	1.2	20.2	249	1.4	10.6	144	2.6	15.2	393
2029	0.7	19.8	142	1.9	10.5	197	2.6	13.1	339
2030	0.4	19.7	80	2.2	10.7	233	2.6	12.1	313
2031	0.4	20.3	88	2.2	10.3	221	2.6	12.0	310
2032	0.5	20.9	109	2.1	10.1	210	2.6	12.3	318
2033	0.9	21.3	190	1.7	10.6	180	2.6	14.3	370
2034	1.1	20.6	232	1.5	10.9	160	2.6	15.1	391
2035	1.3	19.1	248	1.3	10.7	138	2.6	14.9	386
2036	1.4	19.7	286	1.1	10.8	124	2.6	15.8	410
2037	1.1	19.7	219	0.6	10.8	68	1.7	16.5	287
Total	**16.3**	**19.9**	**3,246**	**17.1**	**10.6**	**1,807**	**33.5**	**15.1**	**5,053**

Source: JDS (2019)

The ventilation network will consist of primary exhaust fans located underground. Fresh air will be pulled into the mine workings through the production shaft and through one raise into the base of the open pit. The in-pit raise will supply fresh air to the upper drill horizons while the production shaft will supply fresh air to the lower working levels. The vent shaft will serve as the exhaust route, moving a total of 310 m^3/s through the mine. Level ventilation will be controlled by a combination of regulators, doors, ducting, and auxiliary fans.

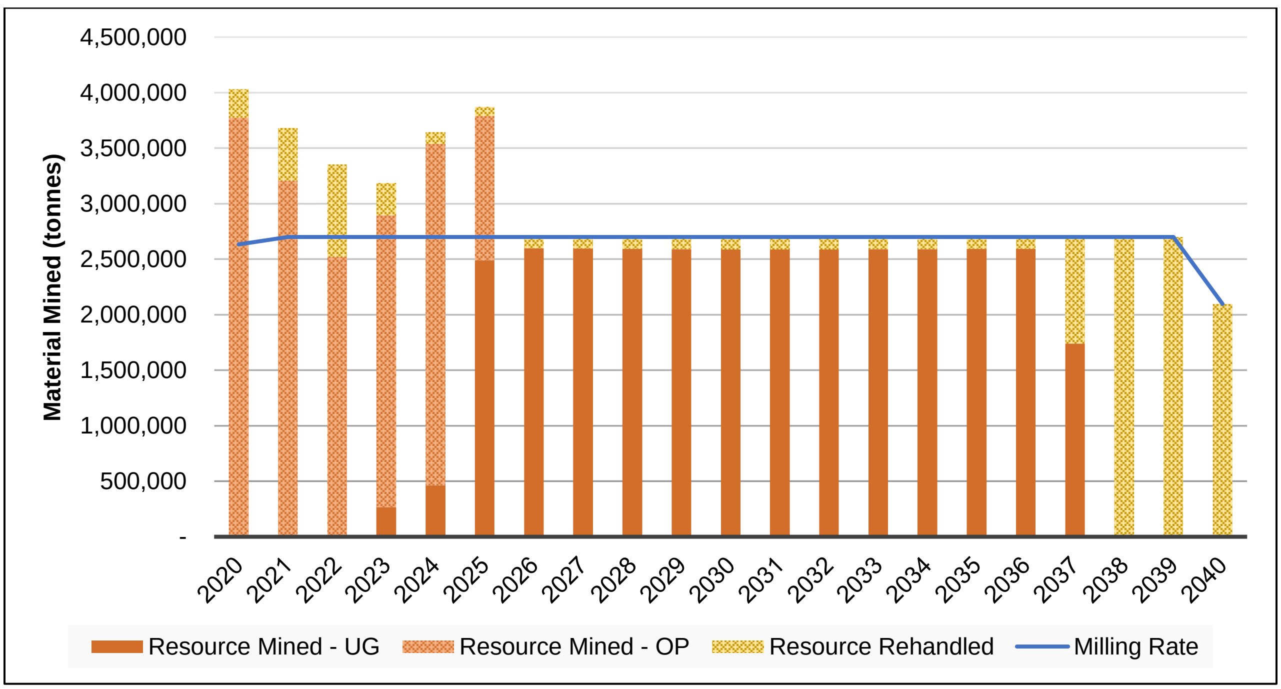
Underground wet-bulb temperatures (WBT) will be maintained below 27.5 degrees Celsius (°C) by employing 6.5 Mega Watts of Refrigeration (MWR) through underground spot coolers. At a coefficient of performance of 3.5, approximately 1.9 MW of electrical power will be required to operate the cooling infrastructure for eight months of the year. During the four cooler months of the year, May through August, mine air cooling will not be required.

Mine and ground water will be collected at the various level sumps and allowed to drain down via gravity to the main pump stations placed at strategic locations in the mine. Generally, there will only be two main pump stations in operation at any time. Pump stations have been designed for a peak inflow capacity of 10,000 m^3/day to handle a 100-300-year storm event and a flood drift has been designed to temporarily store up to 18,000 m^3 of storm water. Average inflow volumes are estimated to be a maximum of 690 m^3/day, reducing significantly near the end of the mine life.

The underground mine will be contract developed and owner operated. Contractors will be utilized for shaft sinking, pre-production lateral development, and raisebore development. Applicable existing open pit employees will be trained during pre-production to transition to the underground mine as the open pit winds down and underground production ramps up. Underground operations will transition to a full owner's team by the time underground commercial production is achieved in 2025. Total underground workforce required per day (day shift + night shift) will peak during pre-production at 312 persons. During normal drill and blast operations, the labour requirement will be 182, and this will be reduced to 111 during final draw down of the South Lobe.

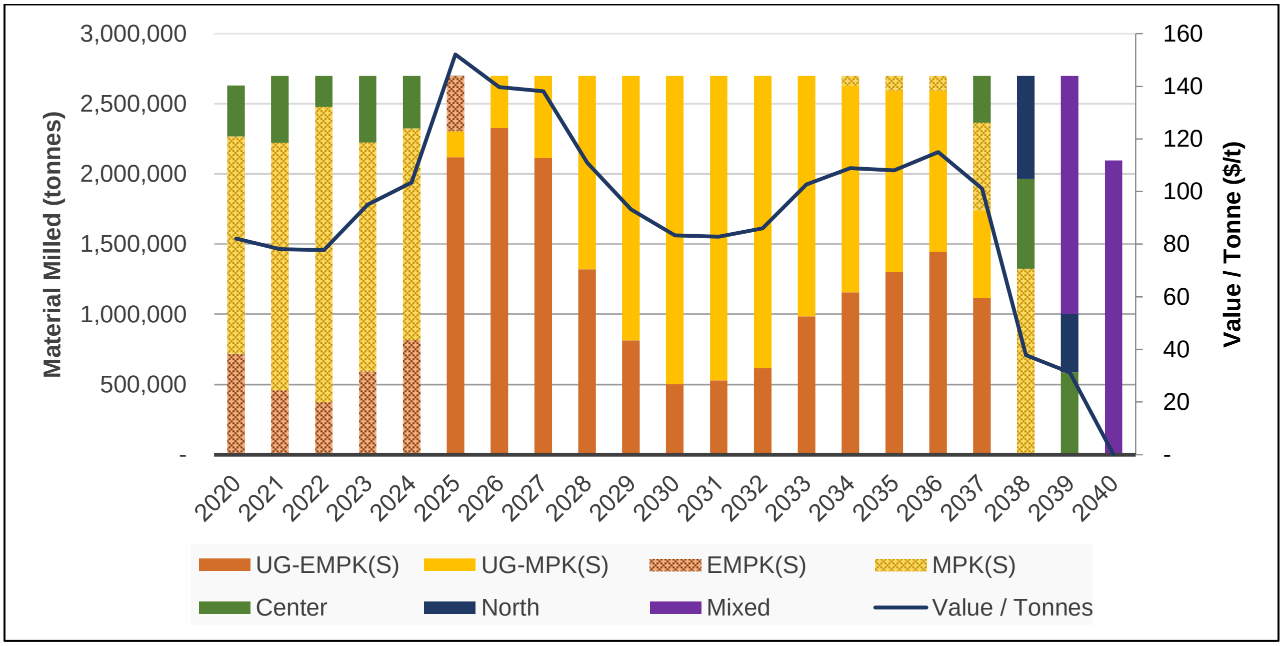
The open pit will continue to operate until mid-2025, overlapping with the underground production for a few months. During the open pit / underground transition, material will be stockpiled or sent to the mill based on processing the highest value ore first. Existing surface stockpiles will be consumed at about 100 kt/y during underground operations and then will be fully exhausted when all mining stops and stockpile processing capacity comes available. The total blended mine and mill feed from both underground, open pit, and stockpile operations is shown in Figure 1-6 and Figure 1-7.

Figure 1-6: Summary of Mine Production



Source: JDS (2019)

Figure 1-7: Summary of Mill Production



Source: JDS (2019)

1.11 Recovery Methods

1.11.1 Karowe Plant History

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

The Karowe plant was upgraded in 2015 with the inclusion of XRT machines installed ahead of the DMS in order to recover large diamonds. This upgrade included the construction and commissioning of a new secondary (gyratory) crusher, 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; and
- Upgrade to the XRT sort house.

Upcoming plant improvements to be completed include:

- XRT Replacement / Refurbishment;
- DMS/XRT Floats (i.e. Coarse Ore Stockpile); and
    - Material from the Coarse Ore Stockpile is earmarked for treatment through the Bulk Sample Plant (BSP);
- Recovery Plant Red Area Tails Dump treatment initiative is anticipated for all associated stockpiles (inclusive of all Tertiary Crusher bypassed feed material).

1.12 Infrastructure

The Underground Project at Karowe will include the use of existing and new infrastructure at the Karowe Mine. Project infrastructure is designed to support the operation of a 2.6 Mt/a mine and 2.7 Mt/a processing plant. The underground project will make use of existing infrastructure including the processing plant, site access road, airstrip, pit dewatering pipeline, maintenance facility and bulk fuel storage.

Existing infrastructure to be expanded or upgraded include the potable water plant, sewage treatment facility, site substation and power distribution, coarse residue facility and fine residue storage facility.

New surface infrastructure will be required to support the UG Project during development and production. This infrastructure includes, but is not limited to:

- New power supply line feeding the project site, including a new substation at the connection point to the grid supply;
- Underground area surface substation and power distribution from the existing site substation;
- Camp complex to support the construction workforce;
- Temporary power supply to support construction;

- Change house for underground personnel;
- Infrastructure pads and roadways;
- Surface sediment pond for managing underground dewatering;
- Buildings and facilities to support the operation including:
  - Underground office complex;
  - Lamp room;
  - Line out room;
  - Training and meeting rooms; and
  - Local first aid room.

1.12.1 Power

The main additional surface infrastructure required to support the UG development will be the construction of a new powerline and associated substations, from the existing Botswana Power Corporation (BPC) transmission line. The Karowe UG operations will require additional bulk power, for a total estimated requirement of 27 to 30 MVA, exceeding the existing contracted Notified Maximum Demand (NMD) of 12MVA. Electrical power to the plant will be supplied from the BPC transmission Letlhakane 400 / 220 kV substation source, where a new 132 kV switchyard will be constructed. A 29 km-long, 132 kV powerline will be constructed as the interface between the substation and the project site. The current AK6 33/ 11 kV substation located within the premises of the mine will be expanded to include a 132 / 11 kV switchyard, where power will then be distributed around the project site.

1.12.2 Residue Storage Facilities

The existing Fine Residue Deposits (FRD), or slimes dams, will need to be expanded to accommodate the additional ore that will be processed from the underground mine.

The current deposition method of the fine residue on site is to place the fine residue behind a waste rock impoundment wall. The current facility is divided into four paddocks, and the impoundment walls are raised in phases to ensure there is sufficient capacity for fine residue deposition and to maintain the legally required freeboard on the facility. A spigot operation is used to deposit the slurry into the active paddock and a pool forms towards the centre of the facility. The water is pumped from this point directly back to the plant.

The FRD will be re-designed with the following features:

- *Phase 1 (Existing FRD footprint):* The impoundment wall will be raised to an elevation of 1,042 masl from the original 1,032 masl design elevation. Raising will be done in two 5 m lifts; and
- *Phase 2*: The new impoundment wall, directly south and abutting the existing FRD, will be built to an elevation of 1,042 masl. This will be done in five 5 m lifts. Phase 2 will be divided into two paddocks.

The raises to Phase 1 will allow the facility to meet the deposition requirements up to 2027. Construction of Phase 2 is required to meet the additional needs generated by the development of the UG, and will begin in 2026, providing capacity starting in 2027.

1.13 Environment and Permitting

The Karowe Mine mining license was approved by the Botswana Department of Environmental Affairs. ML2008/6L is 100% held by Lucara Botswana, a company incorporated in Botswana. The Mining License (ML) was originally issued on October 28, 2008 and was updated on May 9, 2011 to increase the area to the current extent. It is valid for 15 years and gives the right to mine for diamonds. This granted common law surface rights over the entire mining license area and the access road for the duration of the mining lease. An Environmental Impact Assessment (EIA) and an Environmental Management Plan (EMP) were submitted and approved in 2008 and 2010 respectively. The site continues to operate under this license and meet all conditions set out in the EIA and EMP. It was updated in 2013 and 2016 in order to comply with the requirements of Botswana's evolving environmental legislation and to address the associated impacts of the expansion of the process plant and bulk sampling plant in 2016.

The mine continues to monitor the following in accordance with the EIA / EMP:

- Air quality by means of a dust bucket and emissions system sampling monitoring points located at key on- and off-site receptor points;
- Groundwater quality by means of an on- and off-site borehole monitoring systems as well as clean / dirty water control infrastructure on site, specifically monitoring potential seepage from the slimes dam;
- Surface water / storm water control infrastructure by way of infrastructure inspections to ensure the containment of mobilized pollutants in the event of spillages or significant rainfall events;
- Waste Management by means of a waste separation bin system and a lined, on-site landfill for non-hazardous waste; and
- Land disturbance and Environmental incidents by means of continuous inspections.

The EMP is currently being updated and will be submitted for regulatory approval in early 2020 to address the impacts related to the Underground Project. The approved EIA included a Social Impact Assessment (SIA) and outlined specific engagement activities and tools for the community relations personnel. The SIA highlighted that economic opportunities associated with the mine's operations and expansion as well as eventual closure are the primary concern for the majority of stakeholders. In order to continue to strengthen the engagement process, a Stakeholder Engagement Plan (SEP) was developed in 2019 which meets IFC Performance Standards. When completed, it will guide stakeholder engagement by the Community Relations personnel at Lucara.

A conceptual mine closure plan for Karowe was incorporated into the original EIA and EMP submitted, and an associated cost estimate was quantified at the time of commissioning in 2010. A detailed Mine Closure and Rehabilitation Plan (MCRP) and associated cost was developed in 2018 for three potential scenarios (Table 1-6).

Table 1-6: Closure Scenario Cost Estimates

Closure Scenario	LOM (M$)
Unscheduled Closure	16.1
Scheduled Closure w/ Concurrent Rehabilitation	16.0
Scheduled Closure without Concurrent Rehabilitation	25.9

Note: Closure estimates were converted from BWP to US$ using an exchange rate of 10.6.
Source: DWA (2018)

The MCRP will be submitted to the Botswana Department of Mines upon completion of the Karowe UG FS.

The Karowe Mine is not expected to require an EIA for the UG Project, however is expected to trigger a separate EIA for the new electrical transmission line.

1.14 Operating and Capital Cost Estimates

1.14.1 Operating Cost Estimate

A summary of operating costs for the site is provided in Table 1-7. The operating costs below represent total LOM costs (including OP).

Table 1-7: Summary of Operating Cost Estimate

Operating Costs	Unit Rate ($/t milled)	Unit Rate ($/carat)	LOM (M$)
Mining	7.77	55.55	435.4
Processing	14.88	106.40	833.9
G&A (General and Administrative)	5.77	41.24	323.2
Total	**28.42**	**203.19**	**1,592.6**

Source: JDS (2019)

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

The main assumptions used to build up the operating costs are located in Table 1-8.

Table 1-8: Operating Cost Assumptions

Item	Unit	Value
Electrical Power Cost (line power)	$/kWh	0.0897
Average Underground Power Consumption	MW	4.8
Diesel Cost (delivered)	$/litre	0.816
LOM Average UG Manpower (Day shift + Night shift)	employees	182

Source: JDS (2019)

The total LOM operating costs for the underground operations are summarized in Table 1-9.

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

Table 1-9: Underground Mining Operating Costs

Operating Costs	Unit Rate ($/t milled)	Unit Rate ($/carat)	LOM (M$)
Mine Development	0.22	1.46	7.4
Production Stoping	2.90	18.79	94.9
Crushing & Hoisting	1.91	12.40	62.7
Mine Maintenance	1.09	7.04	35.6
Mine General	2.18	14.10	71.3
Contingency	0.42	2.69	13.6
Total	**8.72**	**56.48**	**285.4**

Source: JDS (2019)

Mine development includes all lateral and vertical development required after pre-production. The bulk of mine development will be complete before the start of commercial production, with remaining development limited to drill panel development in the upper levels of the mine.

Production stoping includes the costs to drill, blast, and muck stopes. Rehabilitation of draw points and the operation of support equipment for maintenance and material delivery related to production stoping is included in this cost.

Crushing and hoisting includes all labour, equipment, material, and power required to operate and maintain the crusher, conveyor, and two shafts.

Mine maintenance includes all labour, tooling, and power associated with maintaining the underground mobile fleet and mechanical infrastructure (pumps, fans, power stations).

Mine general includes the cost of management and technical services labour not already captured within the site's existing General and Administrative (G&A) budget. Support equipment to deliver supplies and maintain the underground development is included, as well as the power costs to operate all ventilation, cooling, dewatering and auxiliary power loads within the mine.

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

Process and G&A costs are based on historical and forecasted site operating costs provided by the mine. Process costs have been adjusted to account for the increase in apparent power costs ($/kWh) as the peak power demand, and associated demand charge increase with the load from the underground.

1.14.2 Capital Cost Estimate

The capital costs associated with developing and processing the material from the UG project are outlined below. LOM capital costs total US$722 M, consisting of the following distinct phases:

- Pre-production capital costs total US$514 M and are expended over a five-year pre-production construction and commissioning period; and
- Sustaining Capital Costs total US$208 M which include stay in business costs for the current open pit operation, incurred over the underground project period and costs incurred from commissioning of the underground until the end of mine life.

Table 1-10 outlines the capital cost estimate.

Table 1-10: Summary of Capital Cost Estimate for LOM

Capital Costs	Pre-Production (M$)	Sustaining/Closure (M$)	Total (M$)
Mining	321.7	38.1	359.8
Bulk Earthworks	18.8	-	18.8
Process Plant	0.1	87.9	88
Tailings (CRD and FRD)	-	30.7	30.7
Onsite Infrastructure	5.9	-	5.9
Buildings & Facilities	1.6	-	1.6
Offsite Infrastructure	19.6	-	19.6
Project Indirects	47.7	-	47.7
Owner's Costs	46.9	34.0	80.9
Subtotal	**462.1**	**190.7**	**652.9**
Contingency	51.4	17.8	69.2
Total	**513.7**	**208.5**	**722.2**

Source: JDS (2019)

The details of the cost build up and main drivers of total costs are included below. All costs are in US$ unless otherwise specified.

1.14.2.1 Mining

Shaft development, underground development and infrastructure installations were built up from first principals using a mix of existing on-site contractor rates and expatriate contractors. Utilization of a used headframe and delayed purchase of mobile equipment reduced early capital spending. Equipment and consumable costs are sourced locally where applicable.

- Initial Capital:
  - Shaft head frame, sinking, and equipping (1,480 m): $160.1 M;
  - Underground lateral and vertical development (15 km): $70.7 M;
  - Surface and underground electrical distribution: $15.1 M;
  - Surface buildings: $0.8 M;

  - Underground mobile equipment: $27.2 M;
  - Crusher and conveyor: $4.4 M;
  - Ventilation and cooling: $11.9 M;
  - Sumps and dewatering: $9.3 M;
  - Shops, refuge, lunchroom, communications, and other: $6.7 M; and
  - Pre-production operating costs: $15.4 M.
- Sustaining Capital:
  - Shaft maintenance: $4.2 M;
  - Mobile equipment purchase, refurbishment, and replacement: $30.5 M;
  - Ventilation and cooling maintenance: $0.7 M;
  - Pump rebuilds and dewatering system maintenance: $1.3 M; and
  - Shops, refuge, lunchroom, communications, and other: $1.4 M.

1.14.2.2 Bulk Earthworks

Bulk earthworks were built up from first principles, based on existing contractor equipment and labour rates, or from contractor quotes. The primary cost components in this work breakdown structure (WBS) are as follows:

- Initial Capital:
  - Infrastructure pad, access roads and sediment pond: $0.7 M;
  - Dewatering: $18.1 M;
  - Grout curtains: $5.1 M; and
  - Fan drains and UG grouting: $13 M.

1.14.2.3 Process Plant

Minimal changes to the process plant have been identified as part of the UG FS. Sustaining capital costs include all stay in business costs to support the existing process plant and site infrastructure. These costs were estimated using historical and projected stay in business costs provided by Lucara.

- Initial Capital:
  - Additional metal detection: $0.1 M.
- Sustaining Capital:
  - Yearly capital expenditures to support the mill and existing surface infrastructure of $4.2 M / year: $87.9 M.

1.14.2.4 Residue Storage Facilities

The costs to expand the fine residue deposition (FRD) facility to accommodate the additional slimes generated by the UG Project were estimated based on engineered material take offs (MTO's) and existing contractor unit rates. These were validated by a first principles build up from existing contractor labour and equipment rates. The additional capacity to support the UG Project will be required starting in 2027.

- Sustaining Capital:
  - Expansion of FRD facilities to support the increased requirements over the LOM:
    - FRD expansion (four phases of the new paddock): $30.4 M; and
    - Coarse Residue Deposition (CRD) expansion (surface prep of 3 additional push outs): $0.3 M.

1.14.2.5 Onsite Infrastructure

Onsite infrastructure includes the additional utilities and services required to support the UG Project. It includes the costs of new electrical distribution on site, including the new local substation at the UG area. These costs were based on engineered MTO's and contractor budget quotes or recent actuals provided by Lucara for similar work.

- Initial Capital:
  - Electrical distribution and UG substation: $4.9 M;
  - Sewage & water treatment and distribution: $0.8 M; and
  - IT & communications: $0.1 M.

1.14.2.6 Buildings and Facilities

Buildings and facilities include the additional offices & support facilities, security buildings, change house and other surface rooms / buildings to support the operations. These costs were based on contractor quotes for supply and installation and historical site information.

- Initial Capital:
  - Additional offices and support facilities: $0.5 M;
  - Change house: $0.4 M; and
  - Security infrastructure: $0.7 M.

1.14.2.7 Offsite Infrastructure

Offsite infrastructure costs include all the direct construction costs associated with the construction of the new BPC electrical transmission line and associated substations, along with the costs associated with the construction of the contractor's camp. Costs were based on engineers MTO's and contractor quotes.

- Initial Capital:

    - BPC line, Letlhakane substation and site substation expansion (excluding engineering and contractor indirects): $17.2 M; and
    - Construction camp construction: $2.4 M.

1.14.2.8 Project Indirects

Project indirects include services required to support the Project construction. These include the cost of operating and maintaining the camp during construction, along with the temporary power supply (generator rentals and fuel) to support the sinking of the shaft prior to commissioning of the new BPC line. Requirements were built up from first principles based on staffing or power demands, with costing based on contractor quotes.

- Initial Capital:
    - Onsite contract services: $3.9 M;
    - Temporary facilities & utilities (temporary power): $20.8 M;
    - Contractor indirects & freight: $6.5 M; and
    - Temporary accommodations & expenses: $16.5 M.

1.14.2.9 Pre-Production General & Administrative Costs

Pre-production G&A costs include the incremental staffing and costs for Lucara to support the Project during the development phase. These were based on staffing requirements identified by the current site team using existing Lucara labour rates. Engineering and construction management costs were estimated based on engineering quotes and historical project staffing plans. Taxes on consulting services refer to the 10-15% tax levied on out of country consulting services.

- Initial Capital:
    - Owner's costs: $11.9 M;
    - Engineering: $13.0 M;
    - Construction management: $18.4 M; and
    - Taxes on consulting services: $3.6 M.

1.14.2.10 Contingency

Contingency was applied to the capital costs based on the contingency matrix outlined in Table 1-11 Contingency was determined based on experience on similar projects and the level of detail in engineering design and associated pricing and quotes.

Table 1-11: Contingency

Capital Cost Category	Labour	Perm Equipment	Equip	Other
	(%)	(%)	(%)	(%)
Mining				
Mining - Surface Infrastructure	10	10	10	10
Underground Equipment	5	5	5	5
Underground Infrastructure	10	10	10	10
Underground Development	12	12	12	12
Underground Systems	10	15	10	10
Capitalized Underground Production Costs	10	10	10	10
Shaft Sinking and Infrastructure	12.5	12.5	12.5	12.5
Other Capital				
On-Site Development	10	10	10	10
Dewatering	-	-	-	15
Process Plant	10	10	10	10
CRD and FRD & Mine Waste Management	10	10	10	10
On-Site Infrastructure	10	5	10	10
Buildings & Facilities	10	5	10	10
Off-Site Infrastructure	10	10	10	10
Indirect Costs	5	-	-	10
Owner's Costs	10	-	-	10

Source: JDS (2019)

1.15 Economic Analysis

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

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

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

The reader is cautioned that the diamond prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The price of diamonds is site specific and based on many complex factors.

This study analyzed two scenarios: Karowe Underground Only, and Karowe UG + Open Pit (LOM scenario). The results presented in this section below pertain to the overall LOM economics (including OP).

Table 1-12 outlines the LOM summary and the basis for the economic analysis.

Table 1-12: LOM Scenario Summary

Parameter	Unit	Value
Ore Processed	Mt	56.0
Mill Average Daily Production	kt/d	7.4
Mill Average Annual Production	Mt	2.7
Average Processing Grade	cpht	13.99
Diamonds Contained	k carats	7,838
Diamonds Recovered	k carats	7,838
Recovery	%	100.0
Initial Capital Cost	US$M	513.7
Sustaining Capital Cost	US$M	208.5
Life of Mine Capital	US$M	722.2

Source: JDS (2019)

The main assumptions with respect to the economic model are listed in Table 1-13. Table 1-14 shows the baseline diamond prices by zone.

Table 1-13: Economic Assumptions

Item	Unit	Value
NPV Discount Rate	%	8
Annual Escalation	%	0
BWP:US$ FX	BWP:US$	10.6
ZAR:US$ FX	ZAR:US$	14

Source: JDS (2019)

Table 1-14: Baseline Diamond Prices

Unit	2020 (US$/ct)	2021 (US$/ct)	2022 (US$/ct)	FS (US$/ct)
North	222	222	222	222
Centre	323	329	349	349
EM/PK(S)	618	705	741	777
M/PK(S)	513	578	604	631

Source: JDS (2019)

1.15.1 Results

The economic results for the Project, based on the assumptions outlined above are presented in Table 1-15.

Table 1-15: Economic Results

Parameter	Unit	Pre-tax Results	Post-tax Results
$NPV_{0\%}$	US$M	2,156.7	1,220.4
$NPV_{8\%}$	US$M	945.3	535.4
IRR	%	N/A	N/A
Payback period	Production years	2.8	2.8

Source: JDS (2019)

The LOM economic model does not calculate a meaningful IRR as capital costs are partially offset by operating revenue during the years they are incurred. An underground specific economic model was developed to evaluate the incremental value provided by the development of the project. In the UG only evaluation, the Project showed pre and post-tax IRR's of 21% and 16% respectively.

The post-tax break-even diamond price for the Project ($0 NPV @ 8% discount rate) is US$414/ct.

1.15.2 Sensitivities

Sensitivity analyses were performed using metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant. The Project is most sensitive to the carat price and head grade, followed by the OPEX and least sensitive to the CAPEX. The results of the sensitivity analyses are shown in Table 1-16.

Table 1-16: Sensitivity Results (NPV @ 8%)

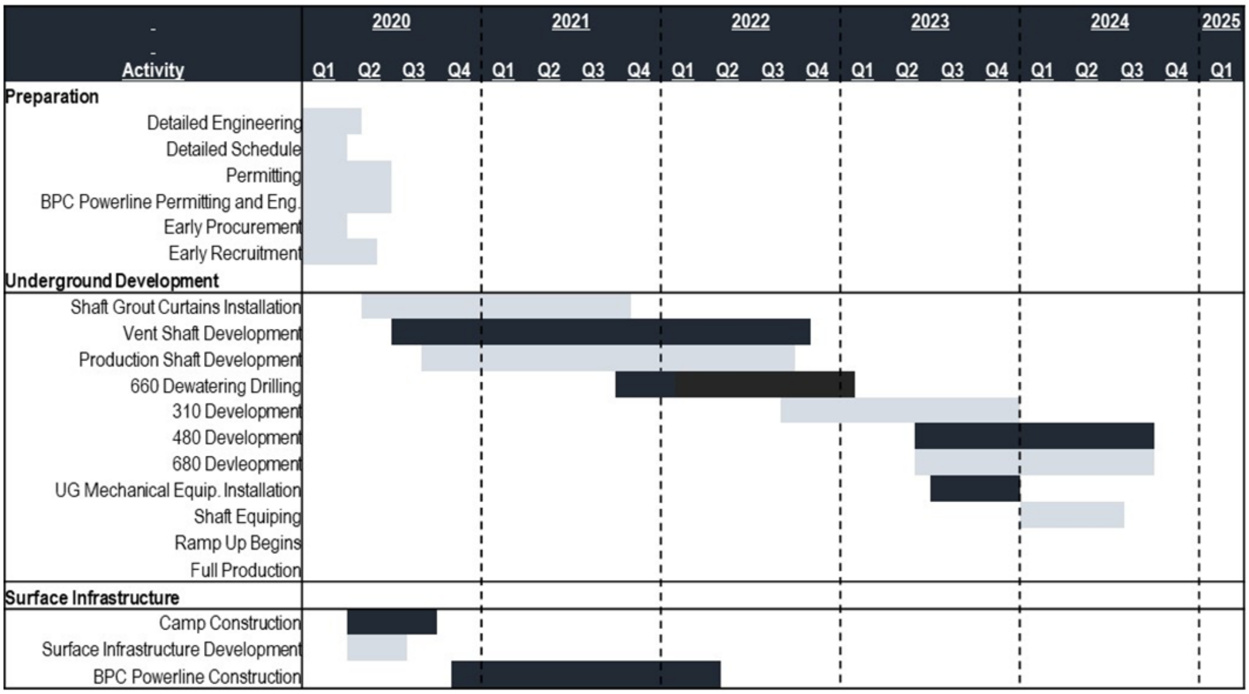
Variable	Pre-tax $NPV_{8\%}$ (M$)		
	-20% Variance	0% Variance	20% Variance
CAPEX	1,046	945	845
OPEX	1,230	945	598
Diamond Price or Grade	474	945	1,417

Source: JDS (2019)

1.16 Project Development

The overall development period for the Project is estimated to be five years from the start of detailed engineering to the underground reaching over 60% production capacity. Activities completed in 2020 will include detailed engineering and permitting, site preparation, camp development and surface infrastructure construction, implementation of the grout curtain and the completion of the pre-sink for the both shafts. Work will continue to ramp up in 2021 as the sinking of the shaft progresses, dewatering activities progress and the BPC powerline is constructed. The shaft sinking will reach the extraction level at the end of 2022, when lateral development will begin. Level development will be complete mid-2024, and production will start to ramp up in Q4 2024, with the underground reaching full production in Q1 2025. Additional details are provided in Figure 1-8 below.

Figure 1-8: Karowe UG Execution Schedule



Source: JDS (2019)

1.17 Conclusions

It is the conclusion of the QPs that the FS summarized in this technical report contains adequate data and information to support a FS study. Standard industry practices, equipment and design methods were used in the FS.

Based on the assumptions used for this evaluation, the Project shows positive economics and should proceed to detailed engineering, financing and construction.

The most significant potential risks associated with the Project are uncontrolled stope back failure, uncontrolled dilution, operating and capital cost escalation, the ability to dewater and depressurize the mine (both OP and UG) ahead of production, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing, skilled contractor and employee personnel availability and diamond price. These risks are common to many mining projects and most can be mitigated with focused engineering, planning and pro-active management. A complete risk matrix is included in the body of this report.

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

1.18 Recommendations

The following early work is recommended. All capital costs that are expected to be spent in 2020 and beyond are outlined in the UG FS CAPEX except for early engineering and procurement initiatives proposed to start in late 2019 as shown below. These early works are estimated to cost US$0.7 M and include:

- Advancing risk mitigation exercises (Dec 2019):
  - Work permits, concrete and local contractor supply;
- Starting value engineering review and optimization of the UG and OP mine plans;
- Starting detailed shaft and mine engineering;
- Starting detailed cost estimation and scheduling with a shaft sinking contractor; and
- Starting procurement on critical path items definition and sourcing.

2 Introduction

This Technical Report was prepared for Lucara Diamond Corp. The report summarizes revised Mineral Resource and Reserve estimates and Feasibility Study for Lucara's Karowe Diamond Mine located near the village of Letlhakane in Botswana. The FS was undertaken to assess the economic viability of an underground mine utilizing resources from the South Lobe of the AK6 kimberlite pipe below the currently operating open pit mine.

The FS and Technical Report were compiled by JDS using guidance from the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. 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 and Mineral Reserves Best Practice Guidelines", November 23, 2003 and "Rock Hosted Diamond Guidance", March 1, 2008. The report describes the incremental contribution of the proposed UG mine as well as the LOM economics of the UG and OP operations combined.

2.1 Qualifications and Responsibilities

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

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

Table 2-1: QP Responsibilities

QP	Company	QP Responsibility / Role	Report Section(s)
Gord Doerksen, P.Eng.	JDS Energy & Mining Inc.	Overall Project Management and Mineral Reserve Estimate	Executive Summary, 1-5, 12, 13.1, 13.2, 13.4, 15, 16.6.1, 20.5, 23, 24, 26-29
Trace Arlaud, Reg. Mem. SME	JDS Energy & Mining Inc.	UG Mining	16 (except 16.3, 16.4, 16.6.1), 21.3.2, 22.2.2
Kelly McLeod, P.Eng.	JDS Energy & Mining Inc.	Comminution	13.3
Carly Church, P.Eng.	JDS Energy & Mining Inc.	Infrastructure, Capital Cost estimate, Owner's Costs	18 (except 18.4 and 18.8), 21 (except 21.3.2), 22 (except 22.2.2), 25
John Armstrong, Ph.D., P.Geo.	Lucara Diamond Corp.	History, Deposit Types, Exploration, Drilling and Sample Preparation, Analyses and Security,	6, 8, 9, 10.1, 10.2, 11, 19

QP	Company	QP Responsibility / Role	Report Section(s)
		Size Frequency and Value Models, Market Studies	
Andrew Copeland, Pr.Eng.	Knight Piésold	Waste management	18.8
Johan Oberholzer, Pr. Eng.	Royal HaskoningDHV	Power Supply	18.4
Matthew Pierce, P.Eng.	Pierce Engineering	UG Geotechnical Considerations	16.3
Markus Reichardt, Ph.D.	Reichardt & Reichardt	Social, Environment and Permitting	20 (except 20.5)
Cliff Revering, P.Eng.	SRK Consulting Inc.	Mineral Resource Estimate	14
Kimberley Webb, P.Geo.	SRK Consulting Inc.	Geology	7, 10.3
Koos Vivier, Pri.Sci.Nat.	Exigo Sustainability (Pty.) Ltd.	Hydrogeological Considerations and Water Management	16.4 & 17.4.9
Lehman van Niekerk, Pr. Eng.	DRA Projects	Mineral Processing	17 (except 17.4.9)

Source: JDS (2019)

2.2 Site Visit

In accordance with National Instrument 43-101 guidelines, all QPs, except Kelly McLeod and Andrew Copeland have visited the Karowe Mine as per Table 2-2. Rather than visiting the mine site, Kelly McLeod visited the laboratory during comminution sample testing. Andrew Copeland relied on site visits by experienced colleagues Justin Teixeira, Mlungisi Motsa and Keneth Matotoka of Knight Piésold.

Table 2-2: QP Site Visits

Qualified Person	Company	Karowe Mine Visit Date(s)	Description of Inspection
Gord Doerksen	JDS	April 18, 2018 December 12-13, 2018 February 18-27, 2019 March 20-27, 2019 April 25-27, 2019 May 14-15, 2019 June 5-11, 2019 July 22-24, 2019	Full review of the operation and discussions with various technical and management personnel.
Trace Arlaud	JDS	May 23, 2018 December 11-13, 2018 December 21-27, 2018	Met with Mining Team, Geologist and Geotechnical Engineers, reviewed the in progress PFS -FS Study, visited the open pit operations – reviewed stratigraphic and kimberlite exposure in pit, visited the core shed, reviewed geology & geotechnical

Qualified Person	Company	Karowe Mine Visit Date(s)	Description of Inspection
			logging, reviewed entire single drill hole from start of hole in waste through stratigraphic sequence into ore, reviewed current hydrology operation, reviewed and discuss geotechnical data acquired to date, examined testing and sampling procedures and reviewed key data analyses required to support feasibility level analysis of mining methods.
Carly Church	JDS	April 25-27, 2019 August 28-September 5, 2019	Review of the operation, and locations of proposed facilities and discussions with various technical and management personnel.
John Armstrong	Lucara	Regular visits since 2013	Full operation reviews of plant, mine and project work including core inspection from any new drilling and analysis of production and sales data.
Justin Teixeira For QP Andrew Copeland	Knight Piésold	December 12, 2018 September 2-3 2019	Project scope, Slimes and tailings operation review, information gathering from various technical/plant personnel.
Mlungisi Motsa For QP Andrew Copeland	Knight Piésold	July 17, 2019 August 1-2, 2019	Information gathering, review of geotechnical site inspection, review of slimes and CRD operations with site personnel.
Keneth Matotoka For QP Andrew Copeland	Knight Piésold	June 26-28, 2019	Geotechnical Investigation supervision for residue facilities.
Johan Oberholzer	RH	October 9-10, 2017	BPC Powerline/Electrical.
Matthew Pierce	Pierce	December 11-13, 2018 February 21-27, 2019	Meet staff and engineers. View the country rock and kimberlite exposures in the open pit. Examine core and log some sections. Review and discuss geotechnical data acquired to date. Examine testing and sampling procedures. Make recommendations for adjustments to geotechnical data collection program. Summarize key data analyses required to support feasibility level analysis of mining methods.
Markus Reichardt	Reichardt & Reichardt	September 9-11, 2017 October 14-18, 2018 December 3-6, 2018	Engagement with site staff and stakeholders to verify EIA, SIA and EMP findings. Examination of site conditions. Examination of consultant procedures to generate monitoring data and findings.
Cliff Revering	SRK	May 14-17, 2019	Review of mine geology, production tracking, mine reconciliation, process plant, geology

Qualified Person	Company	Karowe Mine Visit Date(s)	Description of Inspection
			core shacks and drill core. Discussions with various technical and management personnel. Review of Lucara's Diamond Sales and Marketing Office in Gaborone, Botswana. Inspection of run-of-mine diamond parcel from early May 2019.
Kimberley Webb	SRK	June 11-15, 2018 May 8-17, 2019	Design kimberlite core logging procedure and train geologists. Review of open pit exposures, kimberlite drill core from FS program and geological sampling protocols. Review of Lucara's Diamond Sales and Marketing Office in Gaborone.
Koos Vivier	Exigo	February 20-22, 2018 May 23-26, 2018 May 30-31, 2018 August 13-14, 2018 September 25-26, 2018 November 12-13, 2018 December 3-6, 2018 December 12-13, 2018 February 20-22, 2019 June 4-6, 2019 June 18-27, 2019 October 31 - November 5, 2019	Full review of mine dewatering operations and various meetings with mine specialists related to hydrogeology, engineering infrastructure, drilling, siting and testing. Detailed workshops in Vancouver as well as board meeting presentations in London
Lehman van Niekerk	DRA Projects	September 2-3, 2019	Review of the surface treatment plant process and discussions with various technical and management personnel

Source: JDS (2019)

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 units that are commonly used in industry.

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

Frequently used abbreviations and acronyms are shown in Section 29.

As much as possible, all numbers in this report have been rounded to reflect the appropriate number of significant figures.

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

2.4 Sources of Information

This report is based on information collected by the QPs during site visits, work conducted in 2018 and 2019 including but not limited to information provided by Lucara and other project specialists throughout the course of the FS investigations. Other information was obtained from the public domain. Discussions and data acquisition with Lucara personnel included:

- Lucara data, budgets, plans and schedules;
- Inspection of the Karowe Mine including processing facility, waste facilities, open pit mine, support infrastructure and drill core;
- Review of drilling data collected by SRK and others as part of the FS field program;
- Regional vendors;
- Past internal and external reports, the most recent being the unpublished Royal Haskoning's internal life of mine plan produced at the end of 2018;
- Independent laboratory tests and analyses; 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 information provided by Lucara and numerous internal and external contributors throughout the course of this study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

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

4 Property Description and Location

This section was taken from the unpublished Internal 2018 LOM Report for the Karowe Project, authored by Royal Haskoning and has been amended as necessary for this FS.

4.1 Overview of Botswana

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

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

4.1.1 Types of Mineral License in Botswana

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

- Prospecting Licence: A prospecting license is valid for an initial period of up to 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 Botswana Government paying its pro rata share of costs incurred.
- Minerals Permits: This permit allows companies to conduct small-scale mining operations for any mineral other than diamonds over an area not exceeding a half square kilometre. It is initially issued for five years, with unlimited renewal periods of up to five years each.

4.1.2 Fiscal Regime of Botswana

- The royalty rate on precious stones is 10%.
- There is a negotiated rate of income tax for diamond projects (Section 4.3.2).
- 100% depreciation of capital expenditures is allowed.

- There is a 15% dividend withholding tax on distribution to shareholders.
- Mining equipment and spares are zero-rated, otherwise duties are payable.
- There is 10% Value Added Tax (VAT) which applies to all but zero-rated items and applies to mineral exports.
- There is 15% taxation on revenues for downstream cutting and polishing of diamonds.

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

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

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

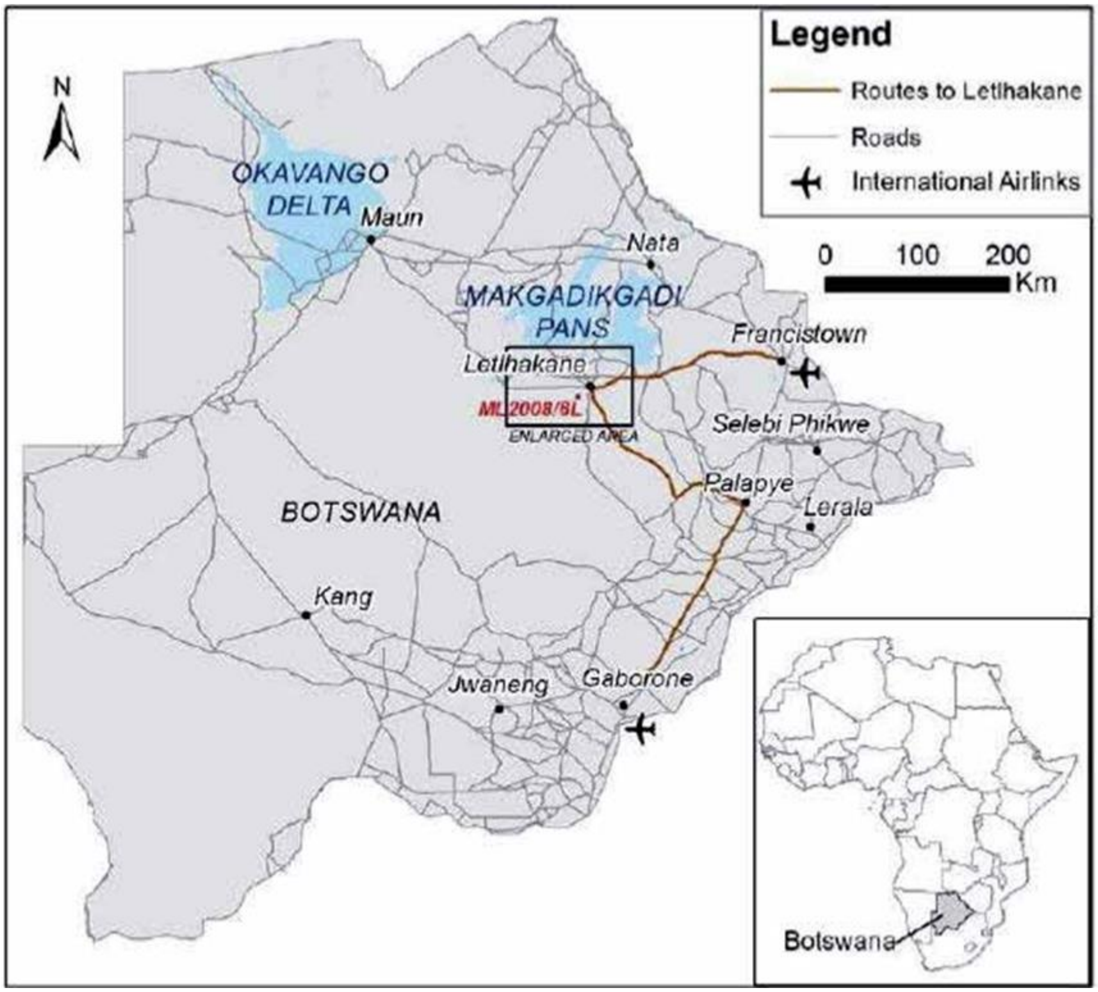
ML2008/6L is 100% held by Boteti, a company incorporated in Botswana. The ML was originally issued on October 28, 2008 and was updated on May 9, 2011 to increase the area to the current extent. It is valid for 15 years and gives the right to mine for diamonds. The Government of Botswana holds no equity in the project. The corner points and geographic location are shown in Table 4-1, Figure 4-1 and Figure 4-2.

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

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

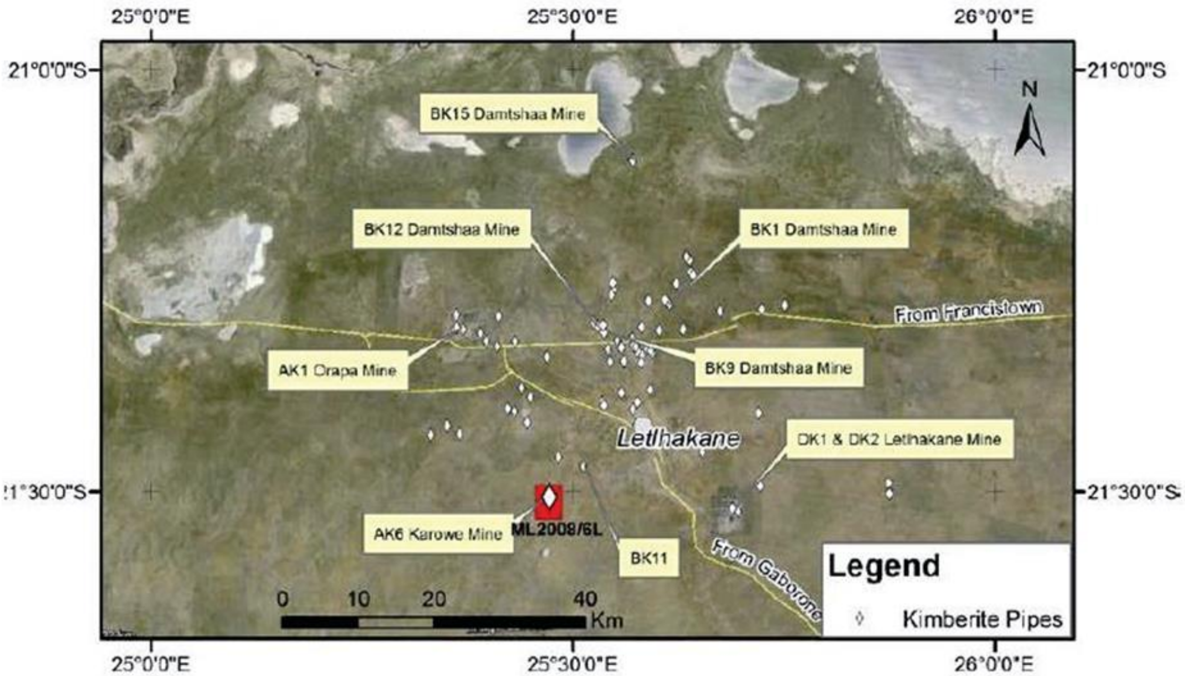
Source: Nowicki et al. (2018)

Figure 4-1: Project Location Map



Source: RH (2018)

Figure 4-2: Project Location Map



Source: RH (2018)

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

Figure 4-3: Aerial View of the Mine Site

Source: RH (2018)

4.3 Permitting Rights and Agreements Relating to Karowe Mine

4.3.1 Surface Rights

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

4.3.2 Taxes and Royalties

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

- 70-(1500 / x) where x is the profitability ratio given by taxable income as a percentage of gross income (provided that the tax rate will not be less than the company rate). Boteti is 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 those to be expected of an active mining operation. These include the open pit, processing plant, infrastructure buildings, a tailings dam, and waste rock storage facilities. The environmental permitting and closure plan is discussed in more detail in Section 20.

4.3.5 Permits

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

Table 4-2: Karowe Diamond Mine Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIA Permit	DEA/BOD/CEN/EXT/MNE 015(7)	EIA valid. EMP updated in June 2016 and will be reviewed to include phase 3 in 2018	Dept. of Environmental Affairs	EIA Act
Water Rights	B6615, B6622, B5386, B 5387, B5388, B5389, B7933B7934, B7935, B7936, B7937, B7937, B7938, B7940, B7941, B7942	Valid for the duration of the mining licence	Dept. of Water Affairs	Water Act
Waste Carriers License	CRLIC/649/06-2080/19 - 002 Kellinicks	20/06/2020	Dept. of Waste Management	Waste Management Act
	CRLIC/649/06-2080/19 - 003 Kellinicks	20/06/2020		

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
	CRLIC/01/12-063/18- SKIP HIRE	31/12/2019	and Pollution Control	
Incinerator Permit	Awaiting certificate from the Department of Waste Management and pollution control	Awaiting department of waste management and pollution control to register and licensing the incinerator	Dept. of Waste Management and Pollution Control	Waste Management Act
Borehole Certificates	In Place	Valid for the duration of the mining licence	Dept. of Water Affairs	Boreholes Act
Dumps Classification	All classified	All dumps active	Dept. of Mines	Mines, Quarries, Works and Machinery Act
Surface Rights	LT/SLB/B/1 IV (231)	09/10/2023	Ngwato Land Board	Tribal Land Act
Radiation License	BW0315/2019	Renewed and certificates will expire in 06 November 2021	Radiation Inspectorate	Radiation Protection Act
Waste Facilities & Sewage Plant	Application in Progress	The mine is working on two projects both at the landfill and Sewage plant to address the findings of the Department of Waste Management and Pollution Control	Dept. of Waste Management and Pollution Control	Waste Management Act
License to manufacture explosives	In Place	31/12/2019	Dept. of Mines	Explosives Act
Permit to carry bulk explosives	F35/13, F34/13 and F36/13	31/12/2019	Dept. of Mines	Explosives Act
Magazine License	386:00002948A and 385:00002947A	31/12/2019	Dept. of Mines	Explosives Act
Blasting License for magazine master	In Place	Valid and appointment renewed yearly	Dept. of Mines	Explosives Act

Source: Lucara (2019)

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

This section was taken from the Internal 2018 LOM Report for the Karowe Project, authored by Royal Haskoning and amended as necessary for this FS.

5.1 Accessibility

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

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

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

5.2 Access

The Property is accessed by 15 km of well-maintained all-weather gravel road from the tarred Letlhakane to Orapa road. Letlhakane village is the closest settlement and offers basic facilities. In 2001, the census noted that Letlhakane had a population of 15,000, rising by 5.7% annually (Central Statistics Office, Gaborone), thus at present, probably has a population of 20,000 to 25,000. There are good telecommunications including cellular telephone networks in the area. Letlhakane is reached from the major cities of Gaborone, Maun and Francistown by good quality tarred roads. There is an 1,500 m airstrip at Karowe, however the closest airport with commercial flights is Francistown, some 200 km to the east and two and a half hours away by road. There is also an airstrip within the nearby Debswana-controlled Orapa Township.

5.3 Local Resources and Infrastructure

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

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

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

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

5.4 Climate

The climate is hot and semi-arid, with an average annual rainfall of 462 mm at Francistown, which falls almost entirely in the summer months from October to April. Summer maximum temperatures are high, generally >30°C, whilst winter days are mild and the nights cold (often <10°C) with occasional ground frost. High diurnal ranges are experienced in all seasons. The climate does not impede mining operations, which can continue all year round. A summary of monthly average temperatures and rainfall are shown in Table 5-1.

Table 5-1: Typical Climate and Rainfall

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

Source: RH (2018)

6 History

The contents of Section 6 are extracted from Nowicki et al. (2018) and Oberholzer et al. (2017) and have been updated as necessary to reflect currently available information.

The AK6 kimberlite was discovered by De Beers in 1969 during part of the same exploration program that between 1967 and 1970 discovered the Orapa kimberlite (named AK1) and the Letlhakane kimberlites (DK1 and DK2). This program also led to a series of other kimberlite discoveries in the Orapa region. Commercial production at Karowe was achieved in July 2012 and has the mine has operated continuously since that date.

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

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

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

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

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

6.2 Debswana Diamond Company (Pty) Ltd. PL 17/86

The current AK6 kimberlite and Karowe Mine lies within former prospecting license PL 17/86 held by Debswana from July 1, 1986 until January 24, 1998. The kimberlite lies within the area dropped at the second relinquishment stage. The primary focus of the work programs on the license was on the discovery of additional kimberlite intrusions, however AK6 was drilled for geological information and to test its diamond content (Debswana, 1999). No details of how it was drilled or sampled are provided, but it was stated as being 3.3 ha in area, comprising hard, dark green kimberlite breccia, and having a diamond grade of 0.42 cpm^3 (approximately 15 cpht; not NI 43-101 compliant).

6.3 De Beers Prospecting Botswana (Pty) Ltd. PL 1/97

PL 1/97 was issued to De Beers Prospecting Botswana (Pty) Ltd. (Debot) on February 1, 1997 and covered the AK6 kimberlite. However, the pipe was within the area dropped at first relinquishment in 2000, and no work was recorded on it.

6.4 De Beers Prospecting Botswana (Pty) Ltd. PL 13/2000

In April 2000, Debot was granted PL 13/2000 with an area of 9.95 km^2 over the AK6 kimberlite. Results from three small diameter percussion boreholes indicated the existence of the North and Central Lobes for the first time. The 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 9 to 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 ML2008/6L 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 M. Government approval which, under the Mines and Minerals Act Section 50 was a condition precedent for this transaction, was given on December 18, 2009. In April 2010, African Diamonds exercised its option to increase its interest by 10.268% at a cost of US$7.3 M. In addition, African Diamonds acquired Wati Ventures and its interest of 1.351% to bring their total shareholding in Boteti up to 40%.

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

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

In December 2011, the AK6 Project was renamed the Karowe Mine and construction of the mine was substantively completed by the end of March 2012. The first production diamonds were recovered in April of 2012. The commencement of full commercial production at the Karowe Mine was declared as of July 1, 2012 and by August 2012 the mine had ramped up to full production.

In November 2012, Lucara recovered a 9.46 ct rare Type II blue diamond at Karowe Mine which it sold for US$4.5 M, and in September 2019, recovered a 9.7 ct Type II blue diamond along with a 4.1 ct gem quality pink diamond. Karowe has established itself as a producer of large gem quality Type II white diamonds as well as a producer of rare gem quality coloured diamonds.

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

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

In September 2017, the Company 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 the Company's ability to recover diamonds prior to the comminution process where diamond damage may occur and thus maximize value for its exceptional diamonds. The Sub-middles circuit allows for diamond recovery down to 4 mm through XRT sensor-based sorting without DMS concentration.

In November 2017, the Company announced the results of its Preliminary Economic Assessment (PEA) for underground development at the Karowe Mine (the "Karowe Underground 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.

Since the onset of commercial production to the end of Q2 2019, the Karowe Mine has produced 2.7M carats (cts) from 17 Mt of processed kimberlite and has sold via tender a total of 2.55 million carats for a total of US$1.5 B resulting in an achieved sold average price of US$686/ct (Table 6-1).

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

Table 6-1: Karowe Mine Production and Sales Results

Year	2012	2013	2014	2015	2016	2017	2018	2019HY*	Total
Kimberlite mined (Mt)	1.6	3.9	3.3	2.4	2.7	1.6	3.1	1.8	20.4
Waste mined (Mt)	4.1	5.5	10.3	11.4	11.1	15.9	15.0	4.3	77.5
Kimberlite processed (Mt)	1.3	2.4	2.4	2.2	2.6	2.3	2.6	1.5	17.4
Carats recovered (Mcts)	0.3	0.4	0.4	0.4	0.4	0.2	0.4	0.2	2.7
Recovered grade (cpht)	22	19	18	16	14	11	14	15	16
Carats sold (Mcts)	0.2	0.4	0.4	0.4	0.4	0.3	0.4	0.2	2.5
Sales average ($/ct)	$274	$415	$617	$612	$824	$847	$502	$463	$586

Source: Lucara (2019)

6.7.1 Significant Stone Recovery to End of Q2 2019

From inception to the end of Q2 2019, a total of 158 diamonds have sold for greater than US$1 M a piece. In the same time period, the Karowe Mine has recovered 14 diamonds > 300 cts, 36 diamonds between 200 and 300 cts and an additional 165 diamonds between 100 and 200 cts.

7 Geological Setting and Mineralization

A detailed account of the geological setting and geology of the Karowe Mine was provided in Lynn et al. (2014). A summarized version was provided in the previous Technical Report (Nowicki et al., 2018) and is restated here, with additional details and updates in Sections 7.3 and 7.4 documenting changes to the geological model, in particular for the deep portion (below ~500 masl) of the South Lobe, based on core drilling undertaken in 2018-2019.

7.1 Local and Regional Geology

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

The country rock at Karowe is sub-outcropping flood basalt of the Stormberg Lava Group, underlain by a condensed sequence of Upper Carboniferous to Triassic sedimentary rocks of the Karoo Supergroup, below which is the granitic basement. The Jurassic (180 Ma) basalts, which are very extensive and underlie much of central Botswana, lie unconformably on the sedimentary succession but are stratigraphically part of the Karoo Supergroup. The regional stratigraphy is shown in Table 7-1. Rocks close to surface are typically extensively calcretized and silcretized due to prolonged exposure on a late Tertiary erosion surface (the African Surface) which approximates to the present-day land surface. There are few outcrops in the Letlhakane area, as the bedrock is concealed by several metres of aeolian sand of the Kalahari Group, reflecting the area's position on the edge of the Tertiary Kalahari Basin. To the south and west of the OKF, the bedrock may be overlain by up to 40 m of Kalahari Group sediments.

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

Table 7-1: Regional Stratigraphy

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

Source: McGeorge et al. (2010)

7.2 Property Geology

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

Table 7-2: Stratigraphic Thicknesses at the Karowe Mine Property

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

Source: modified after McGeorge et al. (2010)

7.3 Kimberlite Geology

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

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

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

The kimberlite in each lobe has been grouped into mappable units (Table 7-3) based on its geological characteristics and interpreted grade potential, including separation of material with very high country rock xenolith dilution (historically referred to as breccias). This is based primarily on extensive drill core logging and core photo review, supported by petrographic studies of representative samples, as well as historical analysis and interpretation of groundmass spinel composition and whole-rock geochemical analysis

(Stiefenhofer and Hanekom, 2005; Hanekom et al., 2006; Tait and Maccelari, 2008; MSC18/005R; SRK, 2019). The main geological features of each unit are summarized below. Unless otherwise stated, the kimberlite terminology and olivine and country rock xenolith size and abundance descriptors used are from Scott Smith et al. (2013, 2018). Note that historical unit names have been maintained for consistency with previous reporting. Minor new units identified in the South Lobe since 2017 are denoted by non-genetic, numbered codes (e.g. KIMB1).

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

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

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

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

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

Source: SRK (2019)

7.3.1 Units Defined by Weathering and Country Rock Dilution

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

Calcretized kimberlite (CKIMB)

The upper parts of all three lobes comprised severely calcretized and silcretized rock. This zone was typically ~10 m in thickness, extending up to 20 m in places. Due to the destruction of textures and resultant difficulty in recognizing specific lithologies within this zone, it was modelled as a separate single unit extending across the top of all three lobes (Opperman and van der Schyff, 2007).

Weathered kimberlite (WK)

The upper 30 to 50 m of kimberlite in each lobe was highly weathered. The intensity of weathering decreased with depth, with fresh kimberlite generally intersected at about 70 to 90 m below surface. Although the primary mineralogical and textural features of the kimberlite were obscured in the upper portions of the weathered zone, this material was seen to transition into the underlying fresh kimberlite units in each lobe. Due to the impact of weathering on the metallurgical properties of kimberlite, separate weathered units were defined in each lobe for those domains where weathered equivalents of the domains were present at surface.

Basalt breccia (BBX/KBBX)

Discontinuous zones of brecciated basalt (BBX), mixed with variable, but generally minor amounts of kimberlite (typically less than 10 %) occur in each of the lobes; they consist of large (meter-sized) to smaller basalt clasts set in a matrix of kimberlite and the majority occur close to the wall-rock contact. An additional unit (KBBX) was defined to encompass kimberlite breccias that are broadly similar to the BBX but display lower levels of country rock dilution (50 to 90 %). KBBX zones appear to be interbedded and/or spatially associated with BBX units. Tait and Maccelari (2008) interpreted KBBX as either talus-type slump deposits or as deposits of possible pyroclastic origin (given their higher kimberlite content relative to BBX). As stated above, these are now largely mined out in the South Lobe but extend below the current mining level in Centre and North Lobes.

7.3.2 North Lobe Kimberlite Units

FK(N) – Fragmental kimberlite

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

7.3.3 Centre Lobe Kimberlite Units

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

CFK(C) – Carbonate-rich fragmental kimberlite

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

FK(C) – Fragmental kimberlite

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

7.3.4 South Lobe Kimberlite Units

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

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

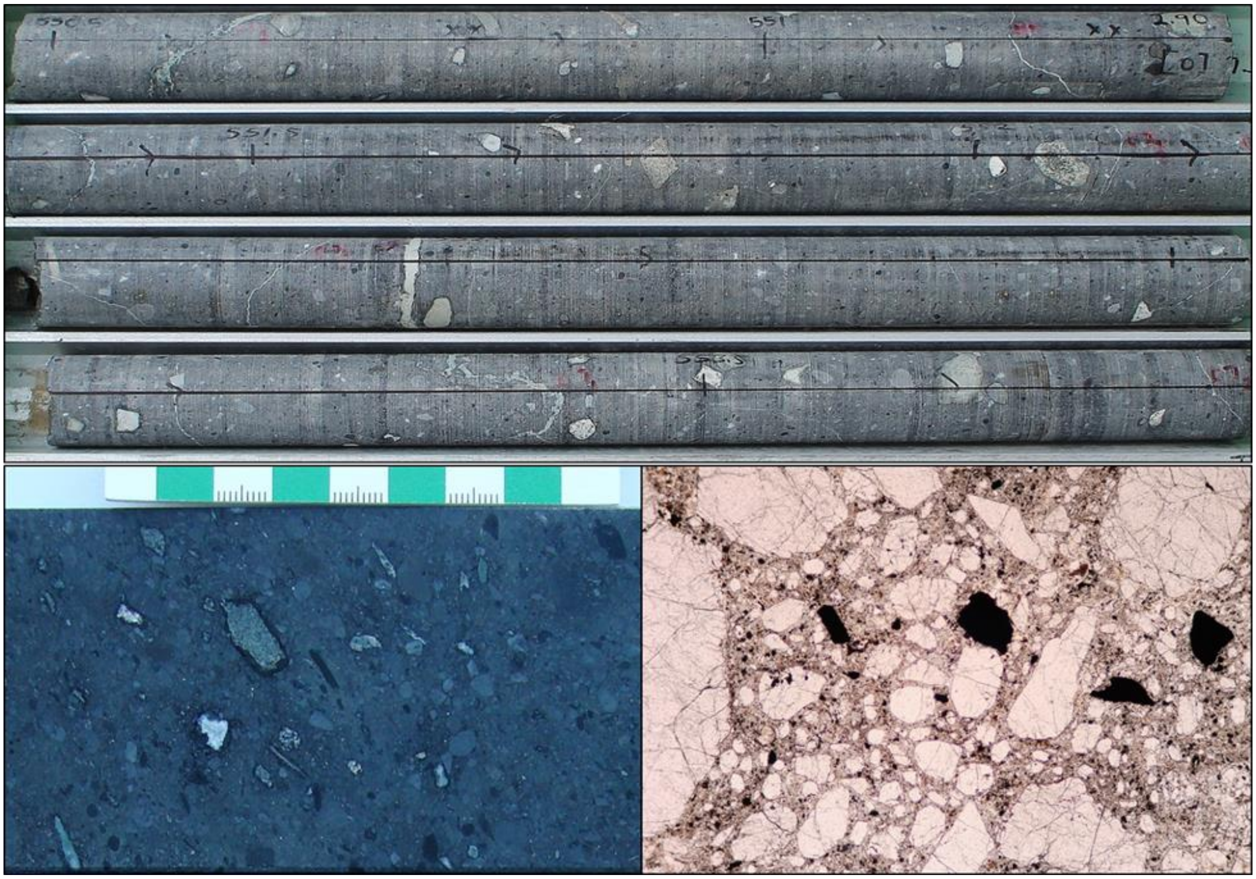
M/PK(S) – Magmatic/pyroclastic kimberlite

M/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly massive to locally crudely-stratified macrocrystic apparent coherent kimberlite. In drill core, M/PK(S) is grey or grey-green in colour and exhibits a 'black spotted' appearance imparted by the presence of common completely kelyphitized (black/brown) garnet macrocrysts and black altered phlogopite macrocrysts. Crude stratification in the form of diffuse fluctuations in olivine and country rock xenolith size and abundance, and preferentially oriented elongate components (such as olivine, small basalt xenoliths, phlogopite macrocrysts) is variably developed. Olivine ranges in size from ultra fine (<0.125 mm) to ultra coarse (> 16 mm) and is predominantly fresh, very abundant (45-50 %) and closely packed. The coarser crystals are inhomogeneously distributed and commonly broken, features atypical of most hypabyssal kimberlite. The groundmass comprises fresh (± serpentinized) monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. A distinct population of thermally metasomatized/ altered country rock xenoliths comprises mainly basalt (as larger grey-green clasts and small <1 cm white elongate shards), lesser (but visually distinctive) white basement granite/gneiss clasts with dark halos and minor Karoo sedimentary rocks. Total country rock dilution is typically low (<10 %), rarely ranging to a maximum of 25 %, and the majority of xenoliths are <10 cm in size. Ilmenite is notably abundant and characterized by variably developed grey reaction rims (comprising fibrous kelyphite-like material). In addition to garnet, ilmenite and rare chrome diopside, M/PK(S) contains orthopyroxene xenocrysts with variably developed reaction rims. The mantle mineral suite includes a distinct population of ultra coarse-grained (> 16 mm, with some up to 5 cm) garnet, ilmenite and orthopyroxene crystals which along with ultra coarse-grained olivine and phlogopite macrocrysts likely belong to the megacryst suite (Schulze, 1987). Peridotite and eclogite xenoliths are present throughout. M/PK(S) is characterized by a relatively high magnetic susceptibility (19 to 30 x 10^{-7} SI).

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

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

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



Note: 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)

EM/PK(S) – Eastern magmatic/pyroclastic kimberlite

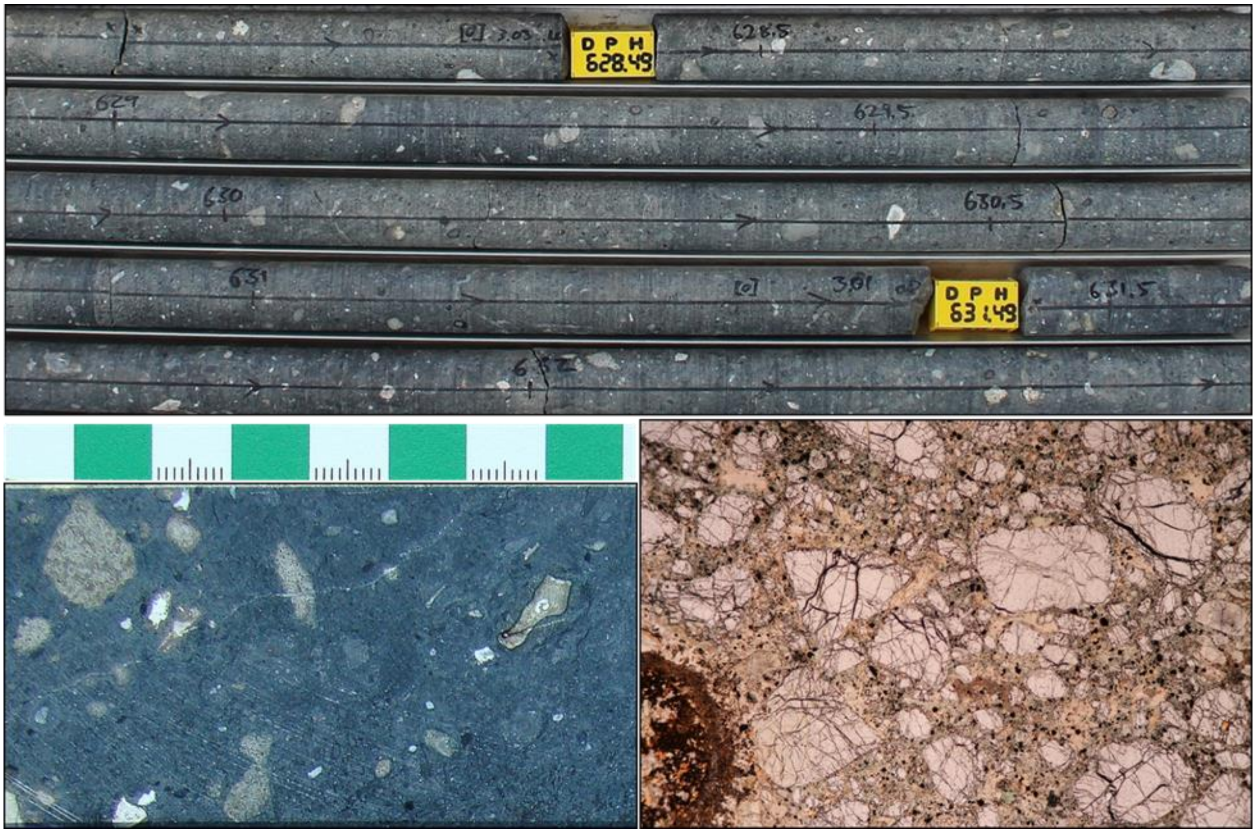
EM/PK(S) is a fine- to coarse-grained olivine-rich, generally country rock xenolith-poor, groundmass-supported, poorly sorted and broadly massive to locally crudely-stratified macrocrystic apparent coherent kimberlite. In drill core, EM/PK(S) is grey-green in colour with variably abundant white 'speckles'. It exhibits a more 'granular' appearance than M/PK(S) due to the olivine being more readily discerned. It lacks the 'black spotted' appearance of M/PK(S) as completely kelyphitized garnet is less common and phlogopite macrocrysts are fresh. Crude stratification in the form of diffuse fluctuations in olivine and country rock xenolith size and abundance is variably developed; preferential orientation of elongate components is rare. Olivine ranges in size from ultra fine (<0.125 mm) to ultra coarse (>16 mm) and is predominantly fresh, very abundant (45-50 %) and closely packed. The coarser crystals are inhomogeneously distributed and commonly broken, features atypical of most hypabyssal kimberlite. The groundmass comprises monticellite, fresh perovskite and spinel, variably enclosed in poikilitic phlogopite plates, and interstitial serpentine/chlorite ± carbonate. Monticellite is typically serpentinized, but the proportion of fresh crystals gradually increases below ~500 masl, and below ~300 masl most samples comprise only fresh monticellite. Groundmass spinel is less abundant than in M/PK(S) and generally occurs as single crystals, with crystal aggregates being comparatively rare or absent. The country rock xenolith population differs from M/PK(S) in terms of the relative proportions, appearance and size distribution of rock types. Basalt is similarly the dominant xenolith type, but it occurs as tan-coloured larger clasts and as a distinct population of small (<1 cm) equant tan or grey-green clasts. Karoo sedimentary rock xenoliths are more abundant than granite-gneiss xenoliths and more commonly exhibit zonal alteration and irregular clast margins. The small (<1 cm) white 'speckles' characteristic of this unit include round carbonate/clay-rich fragments that are possible amygdales derived from disaggregated basalt. The thermal metasomatism/ alteration assemblage of country rock xenoliths in EM/PK(S) includes common clinopyroxene. Total country rock dilution is typically low (<15 %), rarely ranging to a maximum of 25 %, and the majority of xenoliths are < 10 cm in size. As in M/PK(S), ilmenite is characterized by variably developed reaction rims, but its abundance is roughly half that of M/PK(S). Orthopyroxene xenocrysts are more common than in M/PK(S) with less well developed reaction rims. The mantle mineral suite similarly includes a distinct population of ultra coarse-grained (> 16 mm with some up to 5 cm) garnet, ilmenite and orthopyroxene crystals which along with ultra coarse-grained olivine and phlogopite macrocrysts likely belong to the megacryst suite (Schulze, 1987). Peridotite and eclogite xenoliths are present throughout. EM/PK(S) generally has a lower magnetic susceptibility than M/PK(S) (1.5 to 14 x 10^{-7} SI).

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

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

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

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



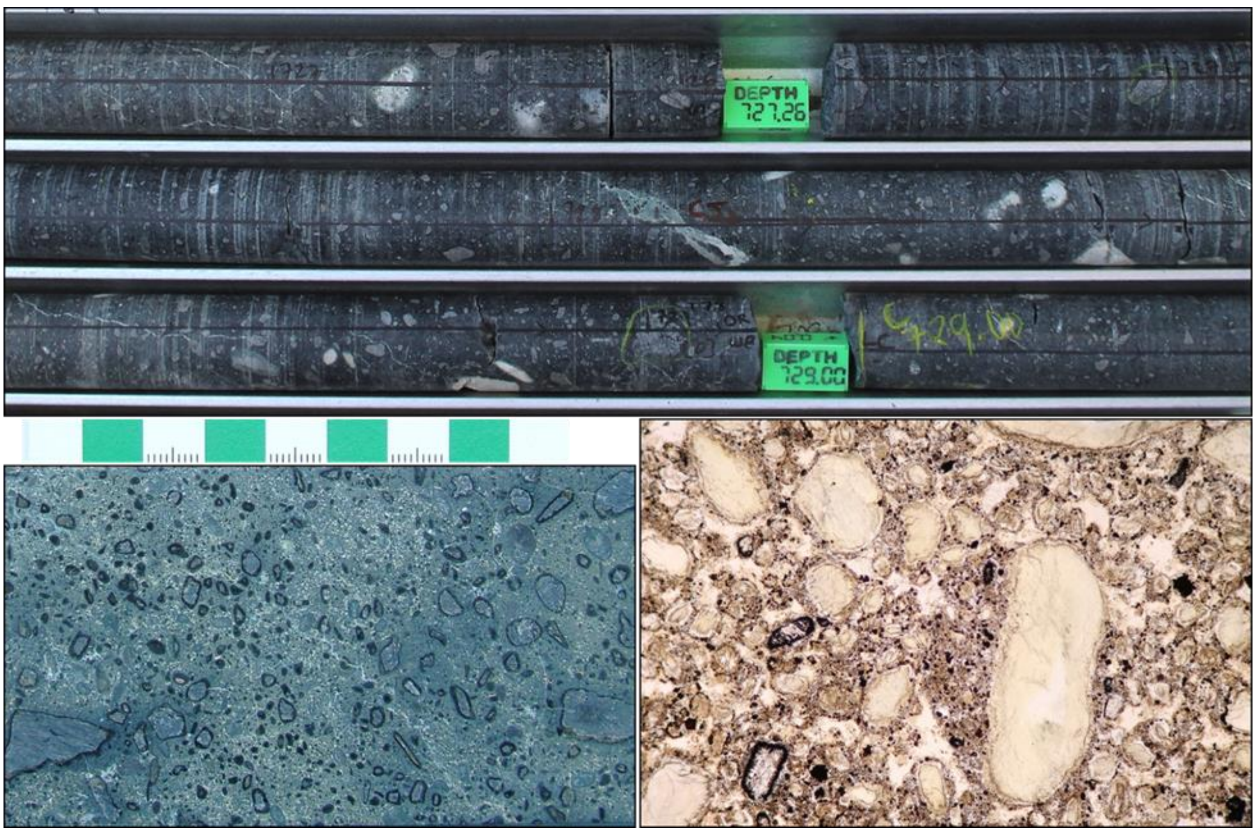
Note: In NQ drill core (top, hole GT001a from 628.0 to 632.5 m), in polished slab (bottom left, hole REP003 at 609.95 m, cm scale) and in photomicrograph (bottom right, hole REP003 at 588.58 m, 20X magnification, PPL, FOV = 7 mm).
Source: Nowicki et al. (2018)

Minor unit KIMB3

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

KIMB3 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor, massive macrocrystic hypabyssal kimberlite. In drill core, KIMB3 is dark grey-green in colour and characterized by readily discernible altered olivine (typically with dark margins) ranging in size to ultra coarse (> 16 mm). Olivine distribution is more uniform than in M/PK(S) and EM/PK(S) and broken crystals are rare. Olivine macrocryst abundance is lower than in M/PK(S), EM/PK(S) and KIMB1. The groundmass displays a variably developed segregationary texture and comprises acicular to prismatic decussate non-pleochroic phlogopite laths, serpentinized monticellite, perovskite, spinel (including common atoll textured crystals), serpentine/chlorite, carbonate and abundant hydrogarnet. Country rock dilution is typically very low (0-2 %) and the xenolith population comprises mainly basalt and granite-gneiss. Garnet is either partly fresh or completely kelyphitized and ilmenite variably lacks or has reaction rims like those observed in M/PK(S) and EM/PK(S). Garnet, ilmenite and mantle xenoliths are generally present in lower abundances than in the other units. Phlogopite macrocrysts are more common than in the other units and are typically completely altered. Autoliths of M/PK(S) and EM/PK(S) and others of unknown origin occur locally. Contacts between KIMB3 and M/PK(S) or EM/PK(S) are diffuse or sharp and finer-grained flow zones are commonly observed at contacts. Well-developed flow differentiation between finer- and coarser-grained components is observed in some intersections. Together these features suggest KIMB3 represents low-volume late-stage sheet intrusions emplaced into the main pipe filling units, possibly in some cases before they were completely consolidated. Magnetic susceptibility readings for KIMB3 are highly variable but in general are the highest of all the units, commonly ranging between 20 and 60 x 10-7 SI. Typical KIMB3 is shown in core, polished slab and photomicrograph in Figure 7-3.

Figure 7-3: Typical Appearance of KIMB3



Note: 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 (2019)

Minor unit KIMB1

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

KIMB1 is fine- to coarse-grained olivine-rich, very country rock xenolith-poor massive to locally flow-aligned macrocrystic hypabyssal kimberlite. In drill core, KIMB1 is dark grey-black in colour with readily discernible mostly fresh olivine ranging in size to ultra coarse (> 16 mm). Olivine distribution is more uniform than in M/PK(S) and EM/PK(S) and broken crystals are present but notably less common. The groundmass comprises abundant phlogopite as ultra fine-grained tablets (which contrasts with the poikilitic plates in M/PK(S) and EM/PK(S) and the prismatic/acicular laths in KIMB3), lesser monticellite, perovskite, spinel, serpentine/chlorite and carbonate. Country rock dilution is typically low (<5 %) and includes basalt, granite-gneiss and Karoo sedimentary rock xenoliths in variable relative proportions. Both fresh and completely kelyphitized garnet are common and ilmenite generally lacks reaction rims like those observed in M/PK(S) and EM/PK(S). Fresh garnet lherzolite and other mantle xenoliths are common. Phlogopite macrocrysts are either fresh or partially altered along crystal margins (leaving the cores fresh). Rare autoliths of unknown origin occur locally. Contacts between KIMB1 and M/PK(S) and EM/PK(S) are typically abrupt yet diffuse in detail, and in rare instances are sharp with finer-grained flow zones. Together these features suggest KIMB1 represents low-volume late-stage sheet intrusions emplaced into the main pipe filling units, possibly in some cases before they were completely consolidated. Magnetic susceptibility readings for KIMB1 are highly variable but most commonly < 20 x 10-7 SI.

Other minor South Lobe kimberlite units

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

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

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

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

WM/PK(S) – Western magmatic/pyroclastic kimberlite

The WM/PK(S) is a pipe-shaped internal kimberlite unit defined in the western portion of the South Lobe that displays geological characteristics apparently different to those of the M/PK(S) and EM/PK(S) units. WM/PK(S) comprises greenish-grey, fine to coarse grained, matrix-supported, poorly sorted, massive apparent coherent kimberlite (historically unclear if magmatic or pyroclastic), and is macroscopically distinct in colour due to its apparent altered character. This material shows additional differences in whole rock geochemistry, percentage DMS yield and rock density relative to EM/PK(S) and M/PK(S). Olivine is serpentinized and locally completely weathered out from drill core. The WM/PK(S) is internally complex, both texturally and in terms of variability in country rock xenolith abundance, which ranges from <10 to 40%. Basalt is the dominant country rock lithology and ranges widely in size from < 1 to > 100 cm. Less common basement and rare black shale xenoliths are also present in places. The geometry of this unit is somewhat speculative due to sparse drill coverage. A possible additional WM/PK(S) intersection was obtained in recent drilling which petrographically is similar to KIMB3, suggesting WM/PK(S) may be the near-surface product of KIMB3 observed at depth, or another similar phase of kimberlite.

7.4 AK6 Geological Model

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

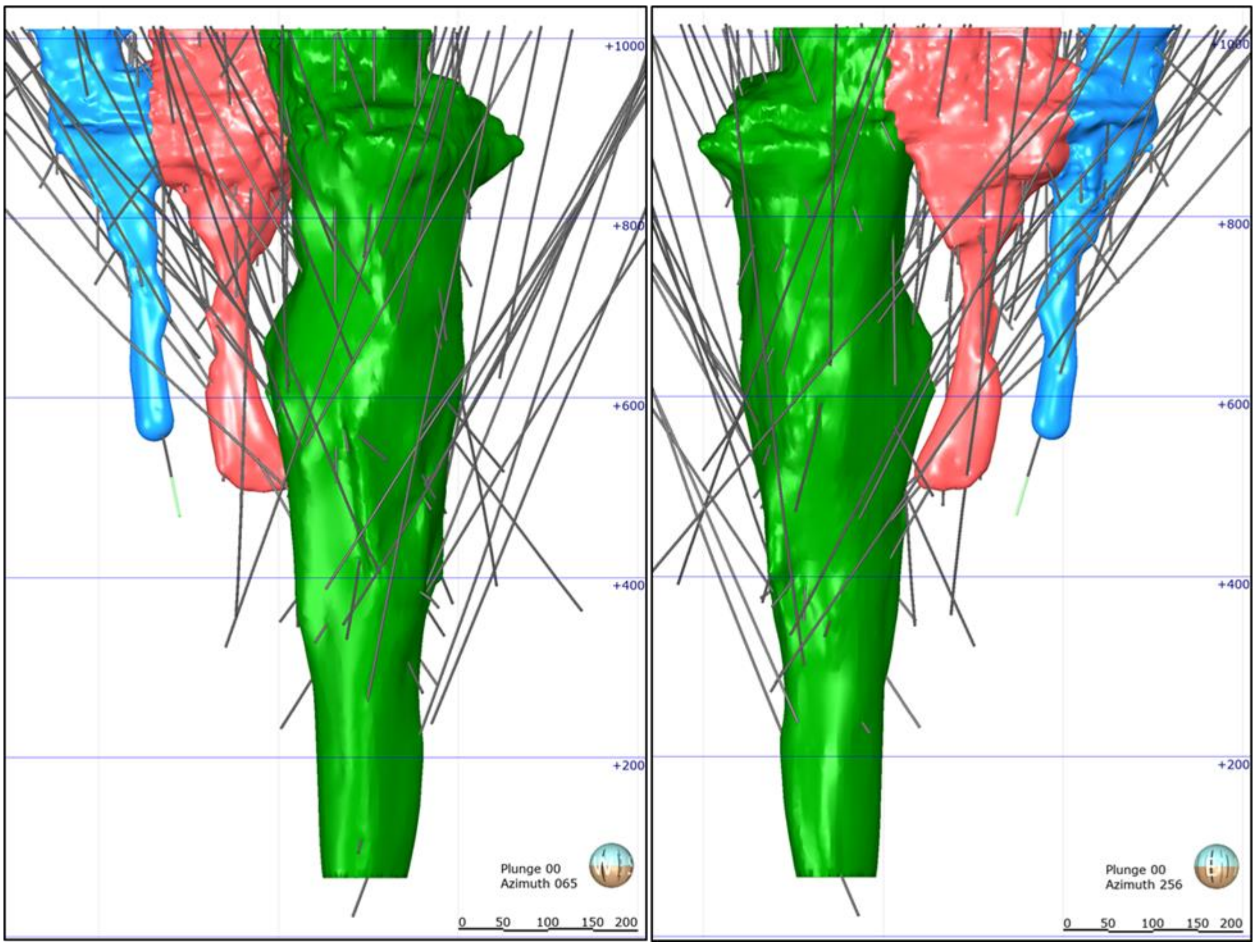
The pipe shell model has been updated (SRK, 2019) from that reported in Nowicki et al. (2018) for recent mining exposure of the contact (all lobes) and at depth in the South, Centre and North Lobes using new pierce points from the core drilling program undertaken in 2018-2019. The base of the South Lobe model has been extended by an additional 190 m. The internal domain model for the South Lobe documented in Nowicki et al. (2018) has been revised (SRK, 2019) based on logging and petrography of the 2018-2019 drill cores. The two main updates are: (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 internal domain model for the Centre and North Lobes remains unchanged from that documented in Oberholzer et al. (2017).

7.4.1 Shell Model

Recent mapping of the external pipe contact defines mining gains in all three lobes and the model has been updated accordingly. In the South Lobe, the data define a pronounced 'bulge' in the pipe margin mainly in the southwest and southeast between 80 and 130 m below surface (920 to 870 masl). This roughly corresponds with the contact between the Stormberg basalt and Ntane sandstone wall-rocks. The downward extent of the gain is constrained by drilling. In the Centre and North Lobes, the volume increases occur from 70 to 100 m below surface (930 to 900 masl) mainly in the east, and are similarly constrained below by drilling.

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

Figure 7-4: AK6 Pipe Shell Model

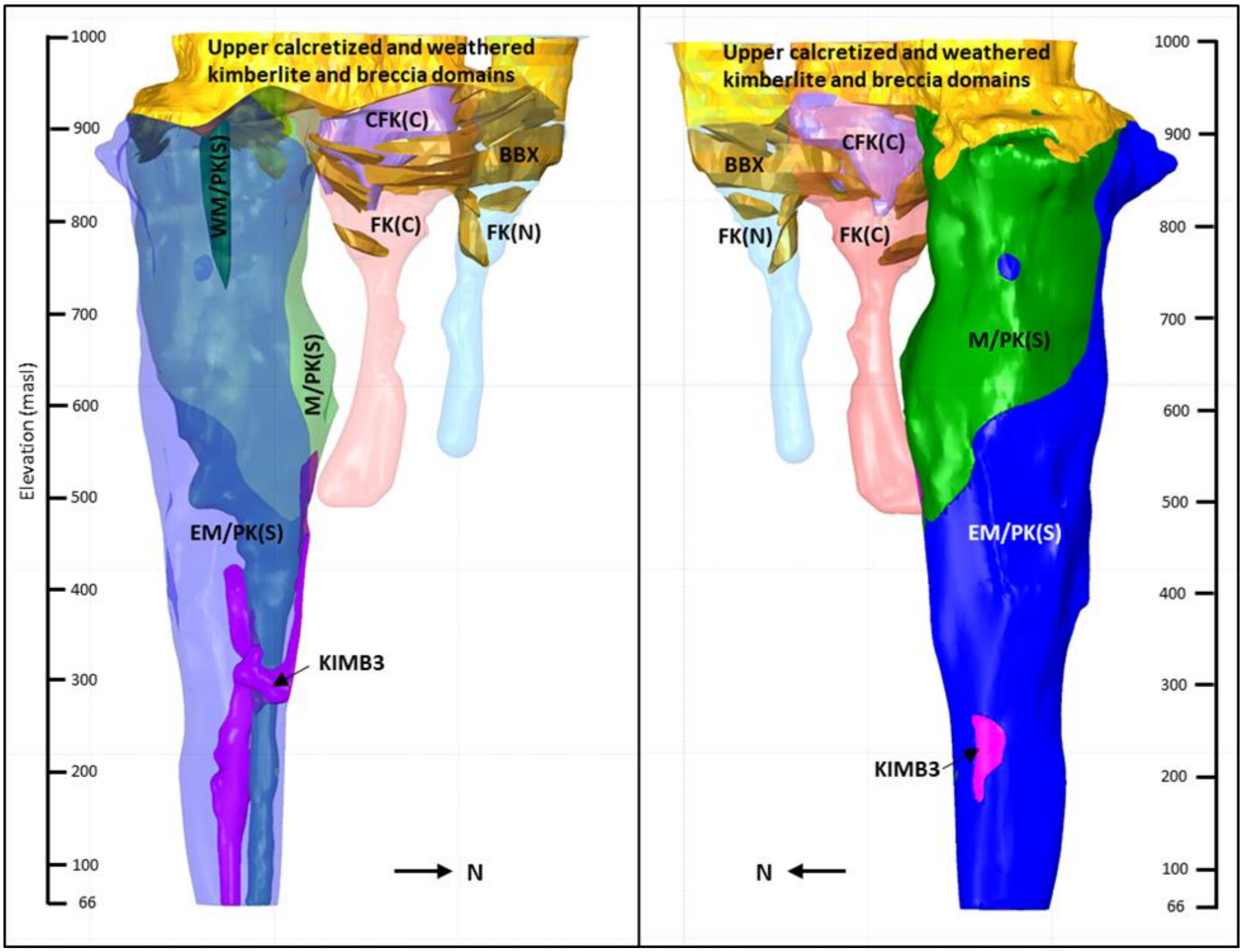


Note: colour coded by lobe (blue = North, red = Centre, green = South) and showing all drill holes (black traces) used to define the model.
Source: SRK (2019)

7.4.2 Internal Domain Model

The internal geological domain model comprises a series of wireframe triangulation solids representing the spatial distribution of the various kimberlite and other (e.g. basalt breccia) units within each lobe (Table 7-3). The internal geological domains are shown in Figure 7-5 and the number and length of core drill holes defining each domain are given in Table 7-4.

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



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

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

Lobe	Domain	Number of core holes	Drill hole intersection length (m)
North	BBX(N), CKIMB(N), WBBX(N), WKBBX(N), WK(N)	13	914.6
	FK(N)	14	1,008.4
Centre	BBX(C), CKIMB(C), KBBX (C), WK(C)	20	1,264.9
	CFK(C)	18	1,047.7
	FK(C)	25	1,272.0
South	BBX(S), CKBBX(S), CKIMB(S), WBBX(S), WKBBX(S), WK(S), IntSWBas	31	2,023.4
	M/PK(S)	52	8,201.3
	EM/PK(S)	44	5,038.1
	KIMB3	7	381.9
	WM/PK(S)	5	341.4

Source: SRK (2019)

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

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

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

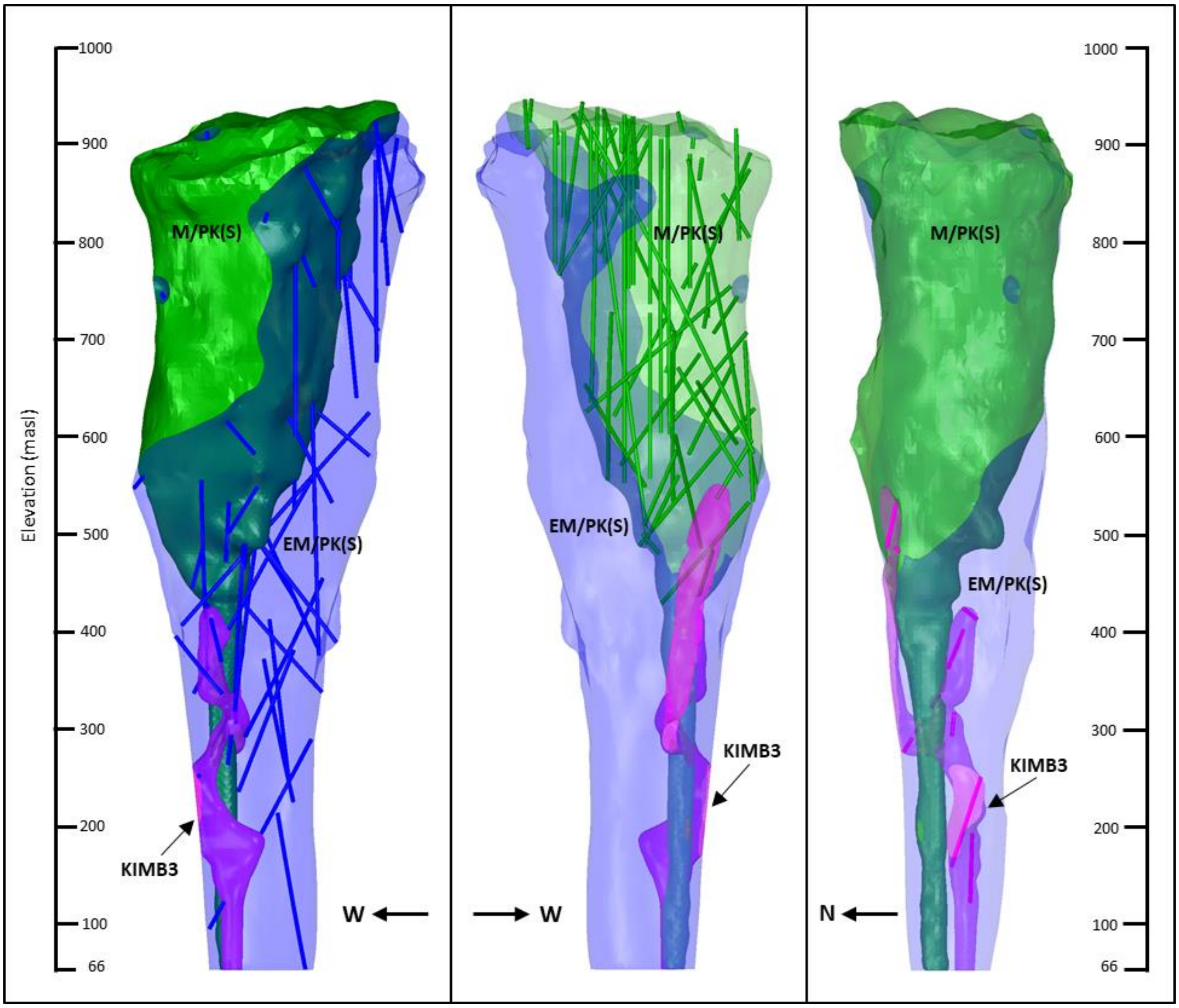
The volumes of the M/PK(S), EM/PK(S) and KIMB3 domains in various depth intervals are shown in Table 7-5. The morphologies of the domains and the internal drill coverage on which they are based are illustrated in Figure 7-6. No changes have been made to the internal domain boundaries reported in Oberholzer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

Table 7-5: Volume estimates of South Lobe internal domains in various elevation ranges (below July 1, 2019 pit surface)

	All domains	M/PK(S)		EM/PK(S)		KIMB3	
Elevation range (masl)	Mm^3	Mm^3	%	Mm^3	%	Mm^3	%
Pit surface (July 1, 2019) to 400	15.19	9.30	61%	5.84	38%	0.05	0.3%
400 to 250	2.02	0.11	5%	1.79	88%	0.13	7%
250 to 66	1.65	0.10	6%	1.41	86%	0.13	8%
Total	18.86	9.50		9.03		0.32	

Note: Due to rounding some columns or rows may not compute exactly as shown.
Source: SRK (2019)

Figure 7-6: South Lobe Internal Domain Model



Note: Looking north (left), south (middle) and east (right) showing the morphology of the M/PK(S), EM/PK(S) and KIMB3 domains (rendered transparent) and the internal core drill coverage used to define them.
Source: SRK (2019)

7.4.3 Geological Continuity

Demonstration of geological continuity within the main kimberlite units is required for the mineral resource estimate to permit (1) assignment of average diamond values derived from production data to kimberlite at depth and (2) assignment of average grade estimates below 604 masl (Section 14). A thorough assessment of the degree of geological continuity was carried out by MSC in support of the resource update reported in Nowicki et al. (2018). This involved review of surface exposures, drill cores and dilution measurements, and an extensive petrographic study. As described in Nowicki et al. (2018) and summarized below, this work confirmed that, with the exception of local variations in the amount of country rock dilution for the

FK(C) and FK(N) units, the main kimberlite units in AK6 are internally broadly homogeneous. Kimberley Webb of SRK carried out much of this work while employed at MSC and has subsequently further assessed the degree of continuity within the kimberlite units based on work conducted since the previous update.

Surface and Drill Core Observations

Historical AK6 geology reports do not indicate any major geological discontinuity with depth within the volumetrically dominant kimberlite units, and grade variations within the units appear to be largely due to locally variable amounts of country rock dilution (Stiefenhofer, 2007; Stiefenhofer and Hanekom, 2005). Kimberlite exposures in the open pit were examined in July 2013, October 2013, June 2017, June 2018 and May 2019. A detailed review of ten complete drill cores was undertaken on site in June 2017, a complete photo review of all 2017 drill cores and of South Lobe historical core photographs was carried out in support of the 2018 update to the geological model, and a detailed review of 13 of the 2018-2019 drill cores was undertaken on site in May 2019. The observations did not highlight any major features or changes in the size and abundance of macroscopic constituents within the kimberlite that would support the presence of a major geological discontinuity within the defined kimberlite units.

Internal Dilution

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

Drill Core Petrography

A large suite of spatially representative petrography samples (n = 227) was collected from drill core in 2017 (92 from historical holes and 135 from 2017 deep drill holes). A further 128 petrography samples were collected from the deep 2018-2019 drill holes. The main objective of the petrographic analysis was to assess the degree of continuity with depth in M/PK(S) and EM/PK(S), the two major units of the South Lobe. Analysis involved the observation of key textural and component characteristics of the samples, including: structure and packing density, olivine abundance and size range, country rock xenolith abundance, type and size, groundmass mineralogy, and kimberlite indicator mineral abundance and types. This work indicated common small-scale variability in these parameters in the M/PK(S) or EM/PK(S), and the presence of a localized potential variant of EM/PK(S); it did not however reveal evidence for large-scale variations or trends in any of these parameters within the M/PK(S) or EM/PK(S) (MSC18/005R; SRK, 2019). Line-scan measurements of olivine size and abundance were not undertaken due to the observed broad-scale homogeneity in these parameters.

7.4.4 Confidence of Geological Model (Volume Estimate)

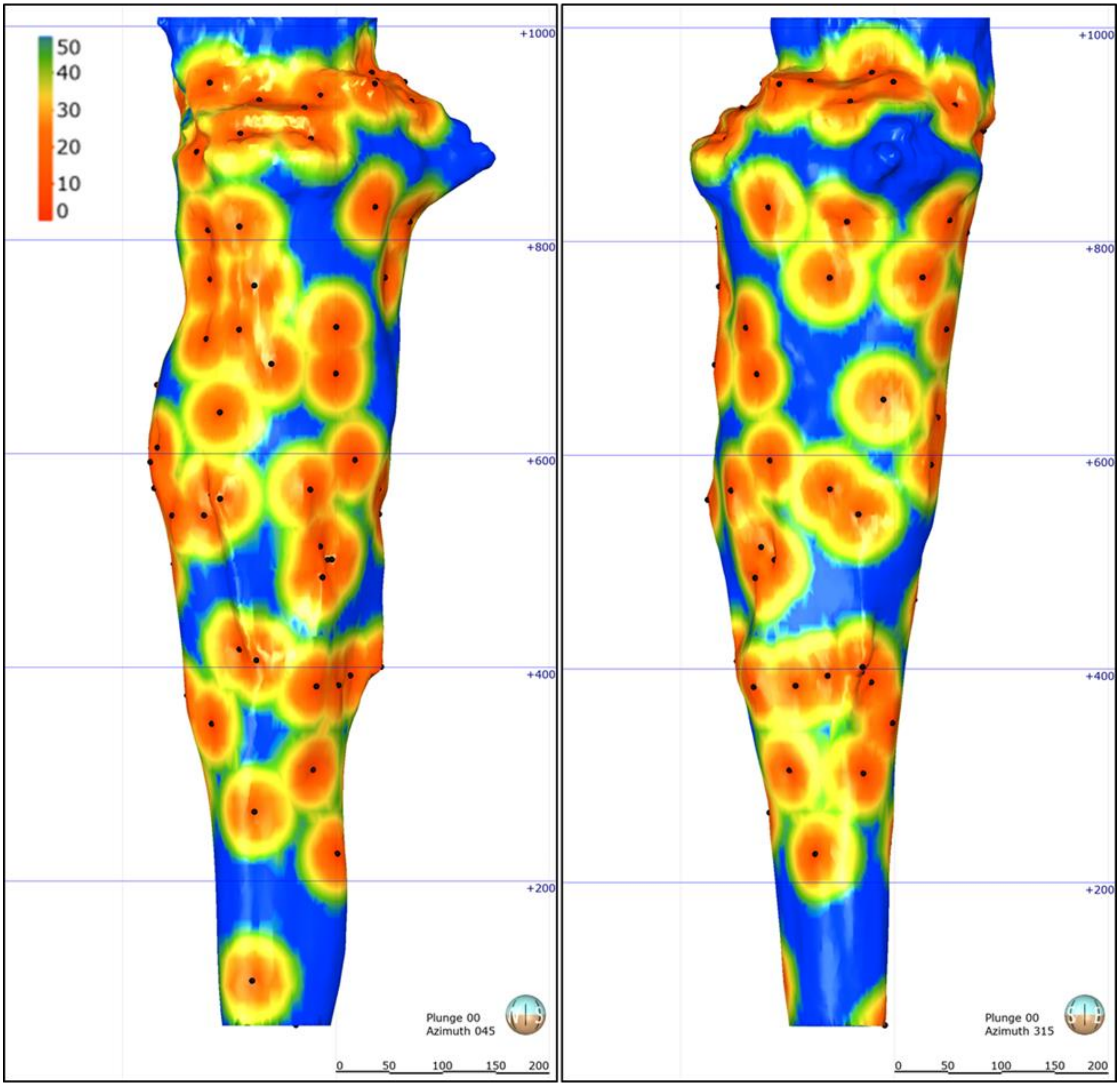
The AK6 pipe shell model is constrained by 182 pierce points from 109 core and LDD drill holes, the majority of which intersect above 600 masl. The model is well constrained in this upper zone by these pierce points and extensive internal coverage providing minimum constraints on the size of the body.

The South Lobe shell model is well constrained by 48 pierce points above 600 masl and by 23 pierce points between 600 and 400 masl. The 2018-2019 drilling provided an additional 14 pierce points in the South Lobe above 400 masl. The model is less well constrained by 12 pierce points between 400 and 250 masl, including six added by the recent drilling. However, while there is scope to modify the exact position of the contact in the gaps between pierce points in this elevation range (Figure 7-7), it is unlikely that the overall pipe volume could deviate by more than ±10 % from the modelled estimate, based on (i) the high degree of confidence with which the shell is constrained above 400 masl and the good continuity with depth in the well-established side-wall dip as confirmed by deeper pierce points, and (ii) the reasonable internal coverage in this elevation range providing minimum constraints on the pipe volume. It is noted that the 20 pierce points added by the recent drilling above 250 masl resulted in <1 % difference in volume between the previous (Nowicki et al., 2018) and current updated models below the July 1, 2019 pit surface (i.e. excluding the mining gains realized between December 31, 2017 and July 1, 2019) and above 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 21,494 m of internal core drilling, of which 15,986 m occurs in the South Lobe. The degree of control on the boundaries between the South Lobe internal domains is relatively high between surface and ~450 masl. There is only a single intersection of M/PK(S) below 440 masl and its volume is thus largely constrained by reasonable internal drill coverage, including intercepts of EM/PK(S) and the newly-defined KIMB3 domain, which confirm where MP/K(S) is not present. The currently modelled distribution of KIMB3 likely represents a minimum volume for this unit.

Nevertheless, the uncertainty in Mineral Resource Estimates below 400 masl noted by Nowicki et al. (2018), which were mostly related to a paucity of drill coverage and corresponding poorer constraints on the pipe shell and internal geology and less representative spatial coverage for microdiamond sampling, has been significantly reduced by the 2018-2019 drilling. The additional drill coverage and microdiamond sampling provide a basis for upgraded confidence between 400 and 250 masl, excluding the KIMB3 domain (as noted in Section 14).

Figure 7-7: Drill Hole Pierce Points in the South Lobe



Note: Drill hole pierce points (black dots) in the South Lobe (left, looking northeast; right, looking northwest) with distance contours. Blue areas are > 50 m from pierce points.
Source: SRK (2019)

7.4.5 Summary and Recommendations

A considerable amount of drilling, geological logging and petrographic work has been undertaken at Karowe 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. Recommendations for further work to increase confidence in key areas include the following:

- Additional drilling and geological assessment of the localized variant of EM/PK(S) in the South Lobe;
- Additional drilling to better constrain the extent of the M/PK(S) domain below 438 masl elevation;
- Additional drilling, geological assessment and sampling of the kimberlite below 250 masl in the South Lobe; and
- Ongoing geological mapping in the open pit of pipe contacts and internal kimberlite domain boundaries.

8 Deposit Types

This section is taken from Nowicki et al. (2018). The primary source rocks for diamonds that are presently being mined worldwide are kimberlites, orangeites and lamproites. All of these are varieties of ultramafic (i.e. Fe and Mg-rich, Si-poor) volcanic and subvolcanic rocks defined by different characteristic sets of minerals. Of these rocks, kimberlites represent the vast majority of primary diamond deposits that are currently being mined.

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

Coherent (previously termed magmatic) kimberlites are the products of direct crystallization of kimberlite magmas, and typically comprise olivine set in a fine-grained crystalline groundmass made up of serpentine and/or carbonate as well as varying amounts of phlogopite, monticellite, melilite, perovskite and spinel (chromite to titanomagnetite), and a range of accessory minerals. While some olivine crystallizes directly from the kimberlite magma on emplacement (to form phenocrysts), kimberlites generally include a significant mantle-derived (xenocrystic) olivine component that typically manifests as large (>1 mm) anhedral crystals. In addition to mantle-derived olivine, kimberlites also commonly contain other mantle-derived minerals, the most common and important being garnet, chrome-diopside, chromite and ilmenite. These minerals, referred to as indicator minerals, are important for kimberlite exploration and evaluation as they can be used both to find kimberlites (by tracing indicator minerals in surface samples) and to provide early indications of their potential to contain diamonds.

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

- Characteristics of the magma (volatile content, viscosity, crystal content, volume of magma, temperature, etc.);
- Nature of the host rocks (i.e. unconsolidated mud versus hard granite);
- Local structural setting;
- Local and regional stress field; and
- Presence of water.

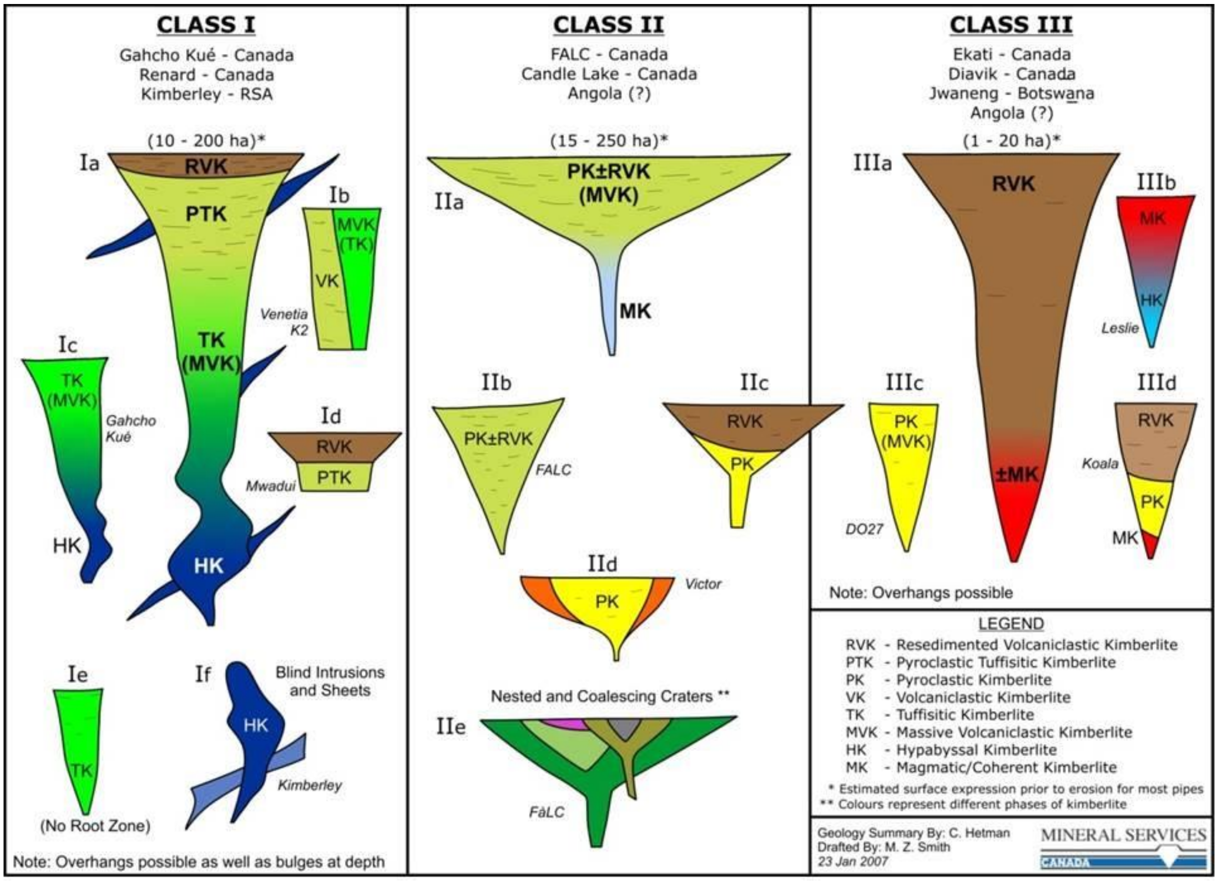
Kimberlites occur at surface as either sheet-like intrusions (dykes or sills) or irregular shaped intrusions and volcanic pipes. The sheets and irregular intrusions are typically emplaced along pre-existing planes of weakness in the country rock. Their emplacement does not involve explosive volcanic activity, and thus they are generally comprised of texturally unmodified coherent kimberlite. In contrast, the pipes are generated by explosive volcanic activity related to the degassing of magma, or the interaction of magma and water, or a combination of both of these processes. This explosive volcanic activity typically produces pieces or clasts of the kimberlite magma (and all the enclosed rock and mineral grains and fragments therein), as well as pieces of the country rock in which it was emplaced. Deposits derived directly or indirectly from volcanic processes which texturally-modify the primary components of kimberlite magma are termed volcaniclastic kimberlite.

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

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

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

Figure 8-1: Schematic Illustration of Common Shapes for Kimberlite Volcanic Bodies*



***The three classes (I, II and III) represent broad groupings with shared attributes of geometry, size and infill.
Source: Nowicki et al. (2018)

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

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

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

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

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

9 Exploration

This section summarizes advanced exploration work (used to support resource estimates) on the AK6 kimberlite carried out by Boteti Exploration (Pty) Ltd. from December 2003 until the completion of the final geological report in May 2007. All work was carried out by De Beers Prospecting Botswana (Pty) Ltd., the operator of the Boteti joint venture, under PL 13/2000. Details on previous work programs are briefly summarized here (extracted and summarized from Nowicki et al. 2018, Oberholzer et al., 2017) and are detailed in Lynn et al., 2014, McGeorge et al., 2010 and various references therein. Recent exploration completed in 2017-2019 included core drilling and sampling of core material and this is documented in Sections 10.2 and 10.3. The current resource estimate is based on data collected during these programs, incorporating results from mining operations and diamond sales since 2012 (Lynn et al., 2014; Oberholzer et al., 2017, Nowicki et al., 2018).

The AK6 kimberlite was continuously held by De Beers under a succession of prospecting licences from the time of its discovery in 1969, until the Project was acquired by Lucara in 2009. The historical sampling, limited and shallow, had shown that it was diamondiferous, but it was initially thought to be very low grade and relatively small (3.3 ha) and as a result further exploration was not a priority. Subsequent work documented a basalt breccia around and over parts of the kimberlite, which was not fully appreciated early in the exploration history of the resource, and that the resource was previously under-sampled.

9.1 Exploration Approach and Methodology

The exploration of the AK6 kimberlite is shown in Table 9-1. It followed a staged approach, which can be summarized as follows:

- Early Evaluation – prior to the Boteti Joint Venture, in late 2003, De Beers carried out geophysical surveys and drilled five x 12¼" holes, which gave a 97 t (in-situ) bulk sample. This resulted in a sampling grade of ~23 cpht and good quality diamonds. Due to a ten-month lapse between the completion of drilling and the release of the sampling results, De Beers committed PL 13/2000 to the Boteti Joint Venture prior to these encouraging results being known.
- Advanced Exploration Phase 1 – Based on the initial work, the AK6 kimberlite was declared an "Advanced Exploration Project". The next step was to define an Inferred Mineral Resource and recover 500 cts from 13 large diameter drill holes at 70 m spacing. The external contacts and internal geology of the kimberlite were explored through an extensive program of delineation drilling and high-resolution geophysics.
- Advanced Exploration Phase 2 – The results of Phase 1 merited Phase 2, the objective of which was to define an Indicated Mineral Resource and recover a large diamond parcel, ideally 3,000 cts, to reduce revenue uncertainty. Large diameter drill holes were placed at 50 m centres and trenches were prepared for recovery of the required parcel of diamonds. Further delineation drilling was also completed. Advanced Phases 1 and 2 overlapped in time, due to a decision to fast track the project. Initial conceptual mining studies showed that exploration should extend to 400 m below surface in the South Lobe, and 250 m below surface in the North and Central Lobes. These were considered to be the limits of possible open pit mining based on an initial economic assessment.

- In 2016 and 2017, two core drilling programs were conducted on the AK6 kimberlite. The combined 12,272 m drilled provided additional pierce points and geological information for the deeper portion of the South Lobe.
- In 2018 and 2019, a combined geotechnical and delineation drill program was conducted with 35 drill holes for a total metres drilled of approximately 22,000 m. Some drilling was specific to the country rock and several holes were designed to test the South Lobe geotechnical purposes with two holes specifically designed to test the South Lobe at depths below 400 masl.

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

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

Source: Lucara (2019)

9.2 Geophysical Surveys

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

Table 9-2: High Resolution Geophysical Surveys Carried out over AK6

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

Source: Lucara (2019)

10 Drilling

10.1 Historical Delineation and Bulk Sample Drilling

Early drilling (2003 to 2007) of the AK6 kimberlite is described in detail in a previous Technical Report dated March 25, 2010 (McGeorge et al., 2010) and the references therein. A brief summary is provided here, extracted from Oberholzer et al. (2017). Drilling can be assigned to three main categories:

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

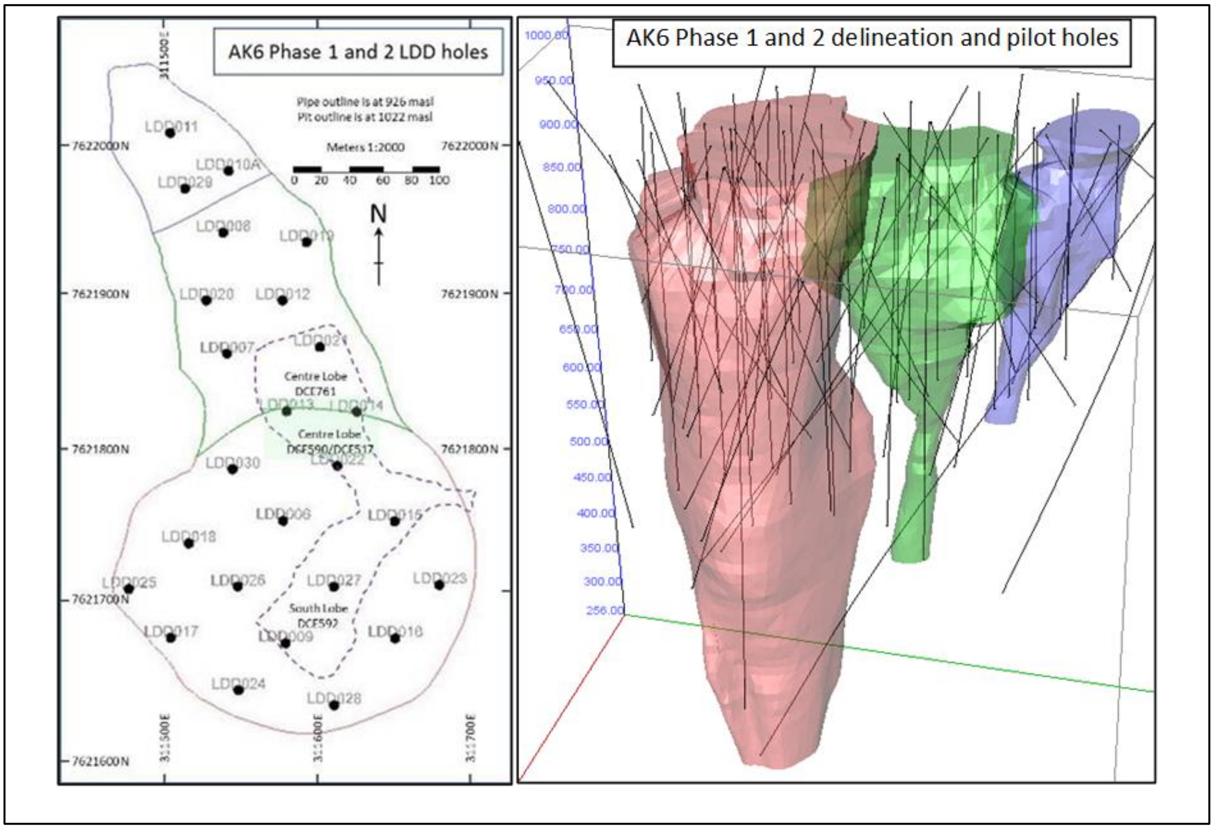
Drilling is summarized in Table 10-1, grouped into the exploration phases described in Section 9 above. Drill hole locations are illustrated in Figure 10-1.

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

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

Source: Lucara (2019)

Figure 10-1: AK6 Phase 1 and 2 Drill Holes



Source: Lucara (2019)

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

10.2 Recent Delineation and Geotechnical Drilling

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

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

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

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

Source: Lucara (2019)

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

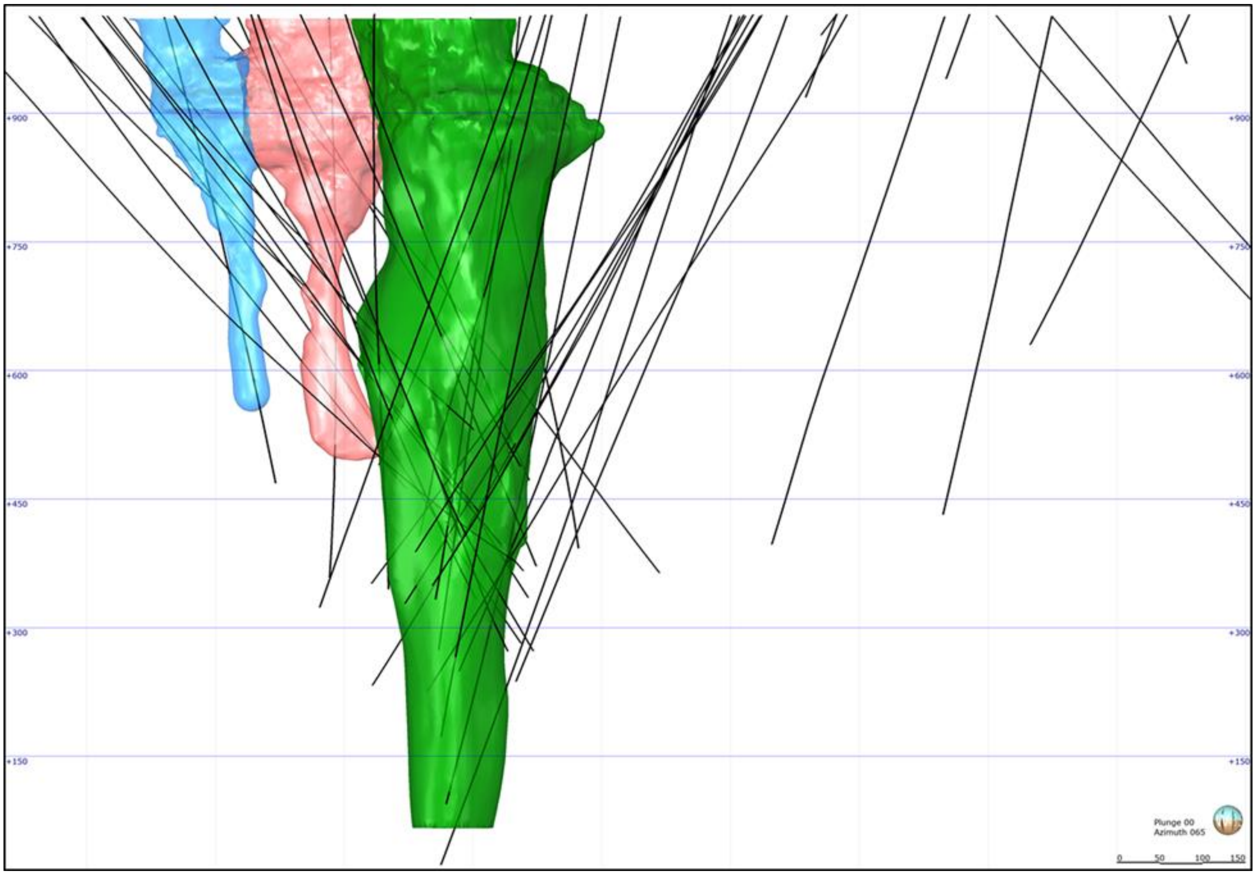
Drill Hole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
CR_GT_DD001	341266	7621936	1013	876	113	-51
CR_GT_DD002	341379	7622174	1012	462	140	-44
CR_GT_DD003	341740	7622103	1012	900	189	-46
CR_GT_DD004	341944	7621869	1012	860	233	-46
CR_GT_DD005	341930	7621517	1013	850	288	-52
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

Drill Hole	Northing	Easting	Elevation (masl)	Length (m)	Average Azimuth	Average Dip
INFRA_GT_DD003	341561	7621357	1014	1,070	19	-68
INFRA_GT_DD004	341352	7621446	1014	903	34	-69
INFRA_GT_DD005	342103	7621197	1013	600	305	-76
INFRA_GT_DD006	341444	7621168	1015	104	335	-69
INFRA_GT_DD006A	341444	7621168	1015	32	269	-51
INFRA_GT_DD007	341548	7621203	1014	969	9	-55
INFRA_GT_DD008	341985	7621696	1013	1,038	270	-62
INFRA_GT_DD009	341452	7621001	1014	81	350	-69
INFRA_GT_DD010	342174	7621078	1014	60	165	-70
INFRA_GT_DD011	341723	7621092	1013	501	168	-47
INFRA_GT_DD012	341446	7620716	1013	429	346	-64
INFRA_GT_DD013	342036	7621166	1013	519	166	-47
KGR_GT_DD001	341413	7622177	1012	698	157	-52
KGR_GT_DD002	341789	7622069	1012	744	210	-45
KGR_GT_DD003	341974	7621820	1013	897	255	-50
KGR_GT_DD003A	341974	7621819	1012	11	253	-54
KGR_GT_DD004	341907	7621480	1013	849	301	-54
KGR_GT_DD005	341627	7621359	1015	615	346	-61
KGR_GT_DD005A	341559	7621629	515	331	350	-58
KGR_GT_DD006	341324	7621487	1013	711	41	-48
KGR_GT_DD007	341224	7621697	1014	800	87	-43
KGR_GT_DD008	341308	7622047	1013	825	139	-51
KGR_GT_DD009	341683	7622141	1012	636	221	-58
KGR_GT_DD010	341852	7622008	1012	800	245	-55
KGR_GT_DD011	341614	7621664	869	604	303	-80
Total				**23,958**		

Source: Lucara (2019)

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

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



Source: SRK (2019)

10.3 Drill Core Sampling

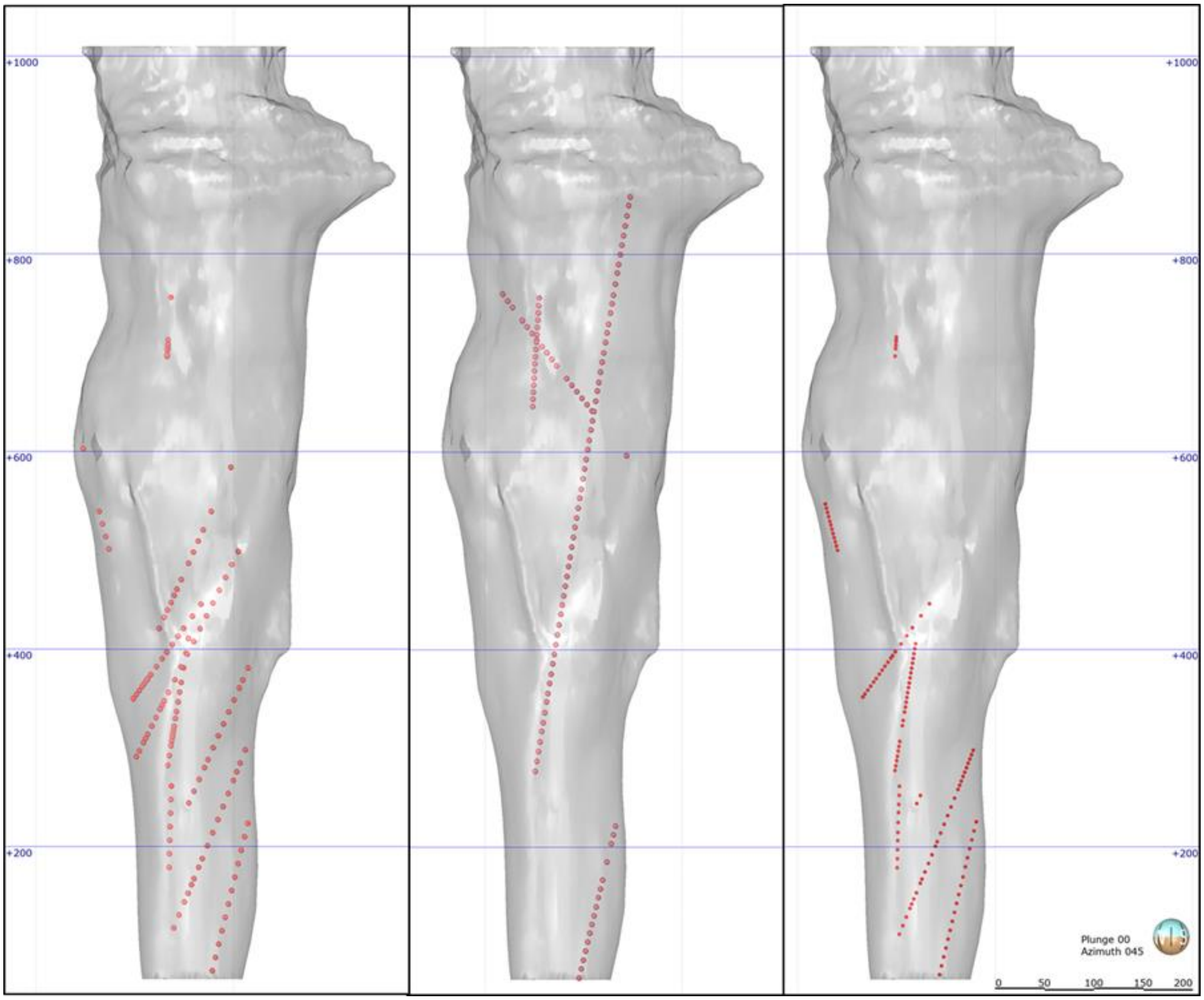
Sampling of drill material in support of historical and recent resource estimates is well documented in previous Technical Reports (McGeorge et al., 2010; Lynn et al., 2014; Nowicki et al., 2018). This section provides details on previously unreported sampling work carried out on the 2018 / 2019 cores (Section 10.2) in support of the updated Mineral Resource Estimate. A key requirement of the estimate is the demonstration of geological continuity within the M/PK(S) and EM/PK(S) units with depth (Sections 7.4.3 and 14.3.5). Sample coverages achieved in the South Lobe are shown in Figure 10-3. Sampling was undertaken for bulk density, petrography and microdiamond analysis, as follows:

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

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

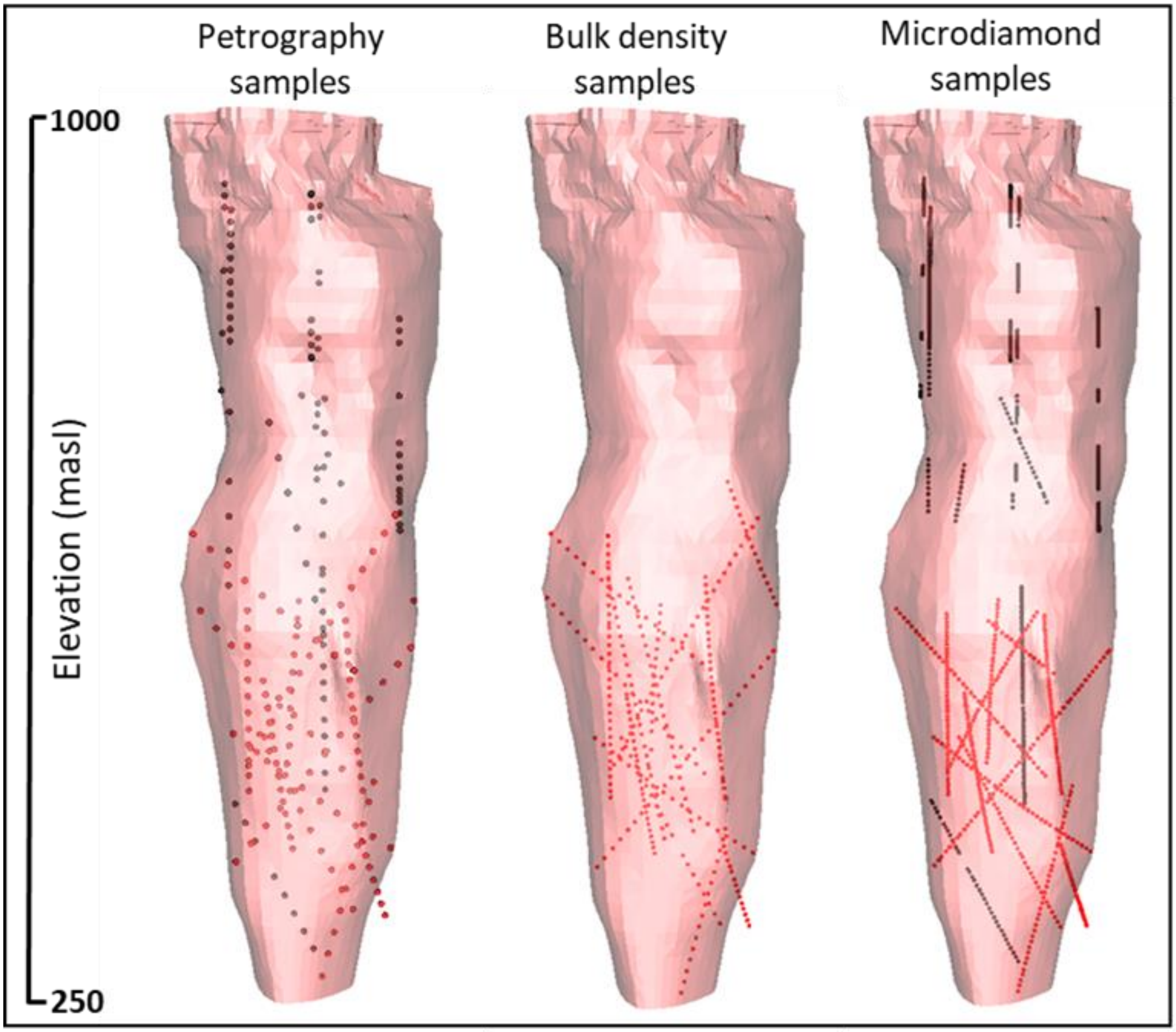
Figure 10-3 shows the locations of 2019 petrography (left), bulk density (center) and microdiamond (right) samples collected from the South Lobe in support of this updated Mineral Resource Estimate. Figure 10-4 shows sample locations for the 2018 resource update (Nowicki et al., 2018).

Figure 10-3: Location of Samples Collected from 2018 / 2019 Drill Core in the South Lobe



Source: SRK (2019)

Figure 10-4: Location of Samples Collected from Drill Core in the South Lobe during 2017



Source: Nowicki et al. (2018)

11 Sample Preparation, Analyses and Security

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

11.1 Historical Samples

11.1.1 LDD Reverse Flood, 23" Drill Samples

These samples were collected during Phase 1 and 2 exploration (Section 9.1) from LDD holes described in Section 10.1. They form the basis of the grade estimate above 604 masl described in Section 14.3.4.

Sample material recovered from drilling was de-slimed to +1.0 mm at the drill using a vibrating screen. The undersize screen was monitored for loss of +1.0 mm material, and if observed, the drill was stopped until the problem was addressed. The sample was collected from the screen in cubic meter sample bags, under the supervision of a geologist. It was then transported to the DMS plant at the De Beers Letlhakane camp by truck, also under the charge of the geologist. At the camp, the responsibility for the samples was passed to the plant foreman. The processing plant was a ten-tonne per hour mobile DMS unit. A total of 4,010 t of +1 mm sample were processed, yielding 306 t of concentrate. The Central and North Lobe concentrate yields averaged 1.1%, while yields from the South Lobe were higher, with averages of between 6 and 8%.

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

Diamond recovery was carried out at GEMDL in Johannesburg. The diamond recovery parameters at GEMDL were the same for all phases. The GEMDL facility was fully ISO17025 certified at the time of sample processing. The recovery area of the GEMDL is a security “red area” and is subject to access control, three tier surveillance and hands-off processing. The concentrates arrived at GEMDL in the same sealed 50 litre drums they had left the sample plant in. Samples weighing 10 kg or more (wet) were treated through the main processing section. Drums within one specific sample were combined to expedite treatment and ease of handling. Material of -4 mm was passed through a dry X-ray sorting process with subsequent magnetic scalping of the X-ray tails to recover non-luminescent diamonds. Material +4 mm was passed through a wet X-ray process with the X-ray tailings dispatched as process tailings.

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

X-ray concentrates were sorted three times, and non-magnetic fractions were sorted once or twice. The sorting efficiency was set at 98% diamond recovery (per carat weight). Recovered diamonds were sent to the final sorting section and stripped concentrate tailings to the hand sort tailings packaging section. A de-falsification process was carried out to remove mis-identified material; where necessary an infra-red spectrometer was used to confirm diamond.

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

11.1.2 Bulk Density Samples

Bulk density measurements were carried out on core samples using a water immersion method, by taking a 15 cm length of core and weighing it in air and in water, drying the sample prior to re-weighing and calculating moisture to derive wet and dry bulk densities (McGeorge et al., 2010). Details of the procedures followed are not available, but the general approach used by De Beers is in line with industry best practice.

11.1.3 Microdiamond Samples

The historical microdiamond dataset for AK6 (77 samples, 1,436 kg) derives from both core and reverse circulation drill chip material. The methods by which these samples were processed, and microdiamonds recovered are not known and the results are not considered reliable (Section 12).

11.2 Petrography Samples

All petrography samples collected in 2017 and 2019 were labelled with the drill hole number, depth and way-up direction by Boteti or Lucara Botswana geologists. No further sample preparation was carried out on site. Petrography samples were shipped to Vancouver Petrographics Ltd. (2017) and Precision Petrographics Ltd. (2019) for processing under the “dry” petrographic sample preparation method. A polished slab preserved with epoxy and two thin sections (standard and wedged) were produced for each sample, for examination under Nikon binocular and petrographic microscopes. Polished slabs, off-cuts and thin sections are in storage at the SRK Consulting office in Vancouver, Canada.

11.3 Bulk Density Samples

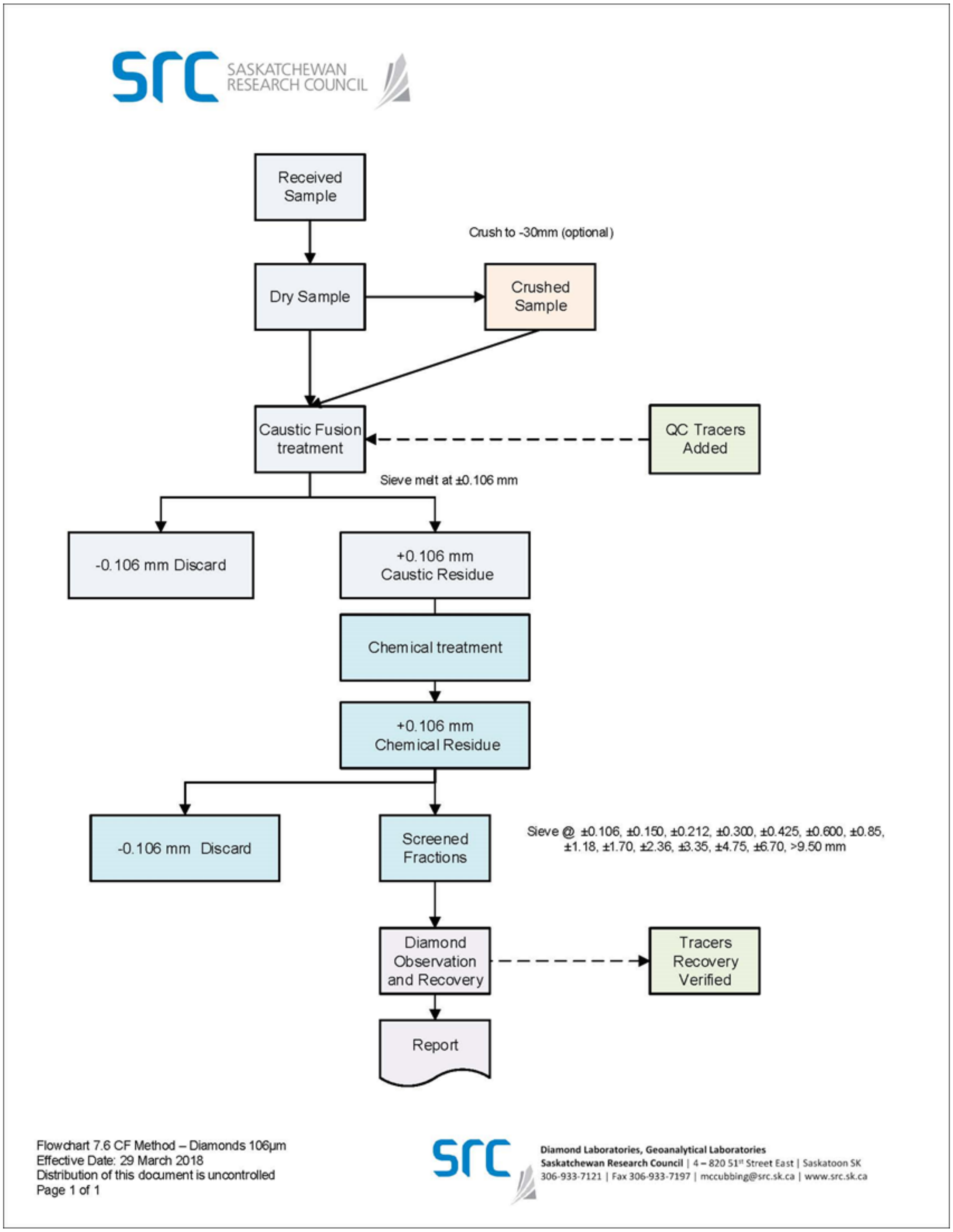
All bulk density sample processing in 2017 was carried out on site by Boteti geologists. Sample masses were recorded at an on-site laboratory and sample volumes were determined by a water-immersion method as per Lipton (2001). No drying of samples was carried out; the bulk density measurements collected in 2017 are not of dry bulk density, and a minor adjustment to account for moisture content (and ensure compatibility between the new and historical datasets) was carried out as documented in Section 12.

11.4 Microdiamond Samples

No preparation of microdiamond samples collected in 2017 and 2019 was carried out on site. Samples of whole core were collected, securely bagged and packaged into 20 L drums for shipping to the Saskatchewan Research Council (SRC) Geoanalytical Laboratory in Saskatoon, Canada. Sample drums were sealed with security tags prior to shipping and the tags were verified by SRC upon receipt. Processing information in this section was provided by the SRC and their process flowsheet is shown in Figure 11-1.

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

Figure 11-1: Processing Flowsheet for Microdiamond Samples Processed at the Saskatchewan Research Council



Source: SRC (2019)

12 Data Verification

12.1 Geological Model

12.1.1 Drill Hole Collar and Orientation Surveys

Early (2005-2007) delineation drill holes 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 recent (2018/2019) drill holes were surveyed by one or more magnetic-based, inertial, or north-seeking gyroscope tools. SRK examined the original and reviewed datasets (following comprehensive QA/QC by Lucara) and concluded the data produced by the EZ-Gyro north-seeking tool were the most comprehensive, reliable and suitable for use in the geological model update. SRK further compared the recent and historical data and no significant issues or discrepancies were noted.

12.1.2 Geological Logs and Internal Geology

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

12.1.3 Internal Dilution Data

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

After review of the drill hole database, including collar and downhole survey data, geological logs, core photos, and internal dilution estimates, SRK 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 current update and estimation of Mineral Resources.

12.2 Mineral Resource Estimate

12.2.1 Bulk Density

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

12.2.2 Microdiamond Data

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

12.2.3 Macrodiamond Data

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

12.2.4 Production and Sales Data

Production and sales data dating back to the start of mining operations in 2012 were provided to SRK as part of the 2019 mineral resource update. Although a detailed audit of this information was not conducted by SRK, the information was reviewed in the context of reconciling past production and diamond revenues with data used for the 2019 mineral resource estimate. No significant issues or discrepancies were noted by SRK during this review.

After review of the microdiamond, bulk sample, and production and sales data for the Karowe diamond mine, SRK is of the opinion that the data is sufficiently reliable to use for mineral resource estimation.

12.3 Mineral Reserve Estimate

Mineral reserve estimates were based on surveyed topography, including stockpiles, along with the 2019 mineral resource block model and detailed open pit and UG mining plans.

Cut-off value estimates were computed based on thorough, first-principle cost estimation for the underground reserves and actual and projected open pit costs for the open pit reserves. Dilution estimates were modelled based on data collected in the extensive, independent geotechnical program conducted in kimberlite and host rock in 2018 and 2019. Processing recoveries were based on actual plant performance and included in the mineral resource estimate. Plant throughput capacity was based on historical results.

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

12.4 Mineral Processing & Metallurgical Testing

12.4.1 Comminution

Regarding comminution data, the first step taken was to review the location of the sample provided by site. Eleven buckets containing rocks from the pit and HQ core from underground were shipped to BaseMet Laboratories in Kamloops, B.C. for comminution test work. The purpose of the test work was to determine if the EM/PK(S) and M/PK(S) material was similar throughout the resource with respect to AG milling. The drill holes used for metallurgical test work 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 underground mine. It is the QP's opinion that there is sufficient data and test work to determine the similarities of the open pit and underground EMPK and MPKS material with respect to AG milling at an FS level.

12.5 Mining Methods

12.5.1 Geotechnical

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

- Inspections of core logging, borehole (wireline) logging, field testing and sampling activities to verify standard operating procedures and associated QA/QC programs
- Logging of selected core and examination of exposures in pit to verify geological origin, thickness/geometry, structural geology and quality of all domains
- Spot intact strength testing (drop, point load and R-hardness) on core and hand samples from pit to verify intact strength of all domains
- Inspection of weathering samples to verify susceptibility of known deleterious materials
- Inspection of blasted kimberlite in pit to verify fragmentation distribution; and
- Spot inspection of borehole collar locations to confirm drilling coverage.

12.5.1.1 Limitations of or Failure to Conduct Verification

Due to scheduling conflicts it was not possible for the QP to directly observe the in-situ stress testing campaign (wireline overcoring). Examination of borehole breakouts from ATV logging were used to estimate the upper limits of in situ stress, which enabled provide partial verification of the in-situ stresses in the kimberlite measured via wireline overcoring. through estimation of the upper limits of in situ stress.

12.5.1.2 Data Adequacy

The volume of data available for the study is considered adequate. The drilling program included completion of 21,837 m of geotechnical drilling from 35 drill holes through both country rock and orebody to support

7,385 field strength (point load) tests and a broad spectrum of laboratory tests encompassing 3,501 total samples. The Total Level of Data Confidence (TLDC) was quantified specifically for the laboratory testing specimens and indicates that the majority of tests met the minimum criteria for the upper limit of the feasibility level study of between 60 - 75%. Lower levels of confidence were obtained for specific thin sub-domains within the Tlapana formation and is related to the small volume of materials available for sampling.

12.5.2 Mining Method and Mining Infrastructure

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

- Onsite Meetings with Technical and Operational staff along with a review of previous studies Prefeasibility and Feasability study;
- Inspection of core shack, logging practices, borehole collars, and material samples to assist in geotechnical verification procedures referenced in Section 12.5.1;
- Inspection of proposed underground entry (shaft) locations to verify offset distances from open pit and other existing and planned mine infrastructure;
- Inspection of site facilities such as workshops, camps, offices, explosives manufacturing and storage, and laydowns to verify areas which can support underground 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 underground production stoping; and
- Import and validation of resource block models to verify mineral tonnes and grade reported in Section 14;

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

12.5.3 Water Management and Hydrogeology

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

- The data gathering was completed or overseen by suitably qualified personnel and reviewed by senior project specialists;
- Data verification was completed by statistical analyses for spatial and temporal data sets;
- Aquifer tests were checked against standard procedures for constant discharge and recovery tests done in the pre-operational phase and packer tests done during the feasibility study.
- Hydrochemical and geochemical tests were completed at accredited laboratories;

- Limitations in data sets were listed and clear recommendations were made to address the gaps;
- Limitations were conservatively accommodated in the modelling and decision-making process so that impacts are over- rather than under-estimated in terms of risks and costs, in line with the precautionary principle.

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

12.6 Environmental Studies & Permitting

The data and information relating to environmental and social aspects of the project were Karowe's original Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) documents cited in the section "Environmental Studies". The veracity and accuracy of the data and information was confirmed in discussions with site staff and through three site visits conducted for this purpose during the course of 2017 and 2018. The information provided in this report is provided without limitations. The qualified person has over 25 years operational, project and corporate experience in this field and based upon this is confident that the information provided is adequate for the purposes used in the technical report.

12.7 Process Description / Recovery Methods

The following steps were taken as qualified person to verify the data reported in Section 17 of the Karowe Mine Underground 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 Karowe projects. The overall Karowe Diamond Mine Block Flow Diagram (BFD) was also verified through previous project engagement(s)/verifications and subsequently amended post site visit early September '19 to confirm recent changes/upgrades. The List of Major Components (summary Mechanical Equipment List for Installed Drives ≥ 100 kW) was verified (reviewed, approved and signed-off) by the client during all implementation phases of the respective Karowe 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.
- No limitations and/or failure to conduct such verification were encountered.

It is this qualified person'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 (Process Description/Recovery Methods) of the document.

12.8 Project Infrastructure & Services

12.8.1 Residue Storage Facilities

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

12.9 Capital and Operating Cost Estimates

Capital and operating costs were built from both current operating experience and first principals using actual regional consumable costs, contractor costs and labour rates. Detailed material take-offs for almost all of the main capital components were estimated.

The information used to generate the capital and operating costs is adequate for a feasibility study.

13 Mineral Processing and Metallurgical Testing

13.1 Mineral Processing Test Work

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

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

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

13.2 XRT Test Work

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

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

Source: Tomra Sorting (2019)

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

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

13.3 Comminution Test Work

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, B.C. Eleven samples in total were received, which included bulk rock samples and drill core from both areas at varying depths. A number of comminution tests on both the bulk and variability samples were completed. The results demonstrated that the two zones, EM/PK(S) and M/PK(S), are similar in hardness with respect to the bulk and variability samples (Doll 2019 and BaseMet 2019).

13.3.1 Sampling

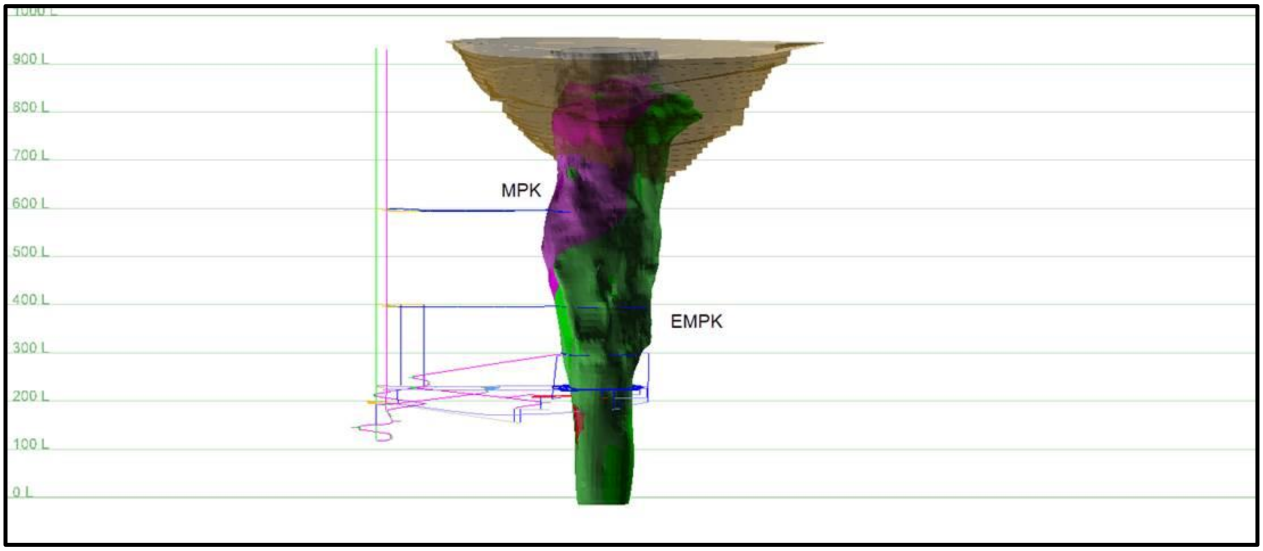
A list of the samples received and the location of the samples are shown in Table 13-1, Figure 13-2 and Figure 13-3.

Table 13-1: Comminution Test Work Sample Selection

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

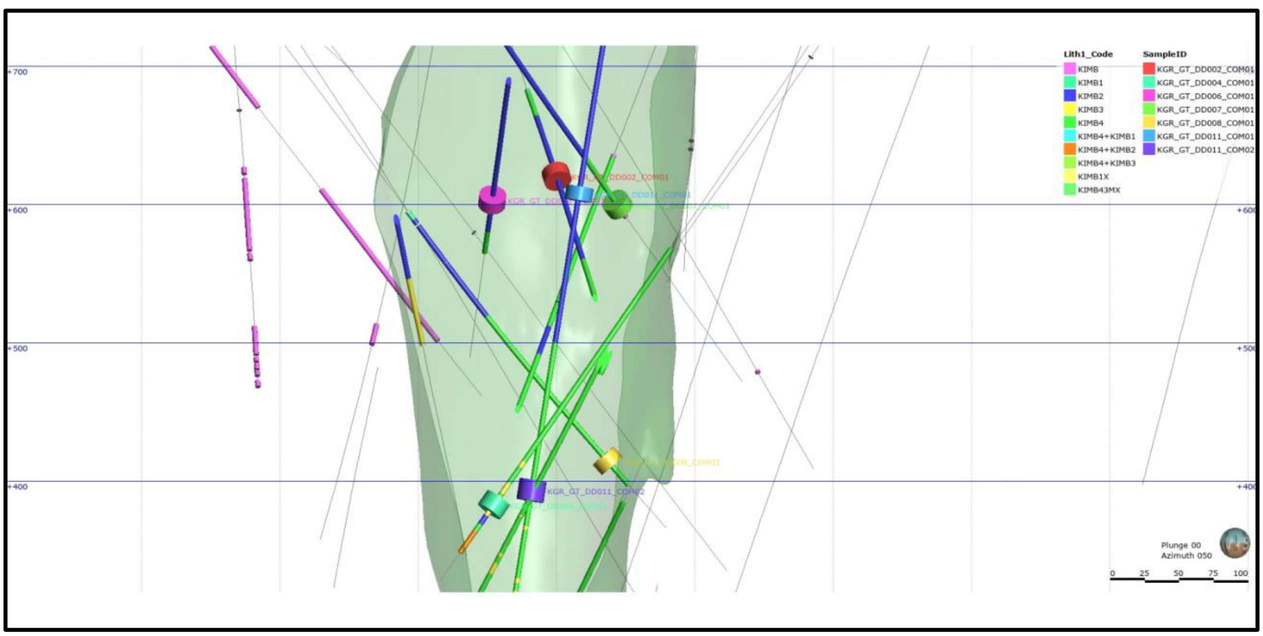
*KIMB3/4 represents EM/PK(S) and KIMB2 M/PK(S)
Source: BaseMet (2019)

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



Source: JDS (2019)

Figure 13-3: Drill Hole Sample Locations



Source: KGR (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 test were completed using the bulk EM/PK(S) and M/PK(S) samples. The results demonstrate 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 was 18.9 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 were in the ranged of 23.7 to 25.1 kWh/t. Both samples would be considered very hard at these size fractions. The JK drop weight test work indicates that the material is moderately hard with Axb values of 38.0 for EM/PK(S) and 43.5 for M/PK(S). The bulk sample test results are shown in Table 13-2.

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

Sample ID	Axb	SG	ta	SCSE	CWi (kWh/t)	RWi (kWh/t)	CSS (µm)	BWi (kWh/t)
EM/PK(S)	37.96	2.96	0.31	10.8	14.2	18.9	300	24.2
EM/PK(S)							212	25.1
EM/PK(S)							150	24.7
M/PK(S)	43.54	2.88	0.30	9.88	17.00	16.8	300	25.1
M/PK(S)							212	24.1
M/PK(S)							150	23.7

Source: BaseMet (2019)

13.3.3 Variability Test Work

Drill core representing EM/PK(S) and MP/K(S) at different elevations in the ore body was collected and composited to create seven different variability samples. The results indicate the SAG Mill Comminution (SMC) and BWi are similar for all samples tested. The RWi ranged from 17.3 to 21.5 kWh/t with M/PK(S) being slightly softer and not demonstrating a significant correlation between hardness and depth. The summary of the variability test work is outlined in Table 13-3.

Table 13-3: Summary of Variability Samples Comminution Test Work

Sample ID	Ore Zone	DWi	DWi	Mia	Mih	Mic	A	b	Axb	SG	ta	SCSE	F80 µm	P80 µm	Gpr	RWi	CSS µm	F80 µm	P80 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

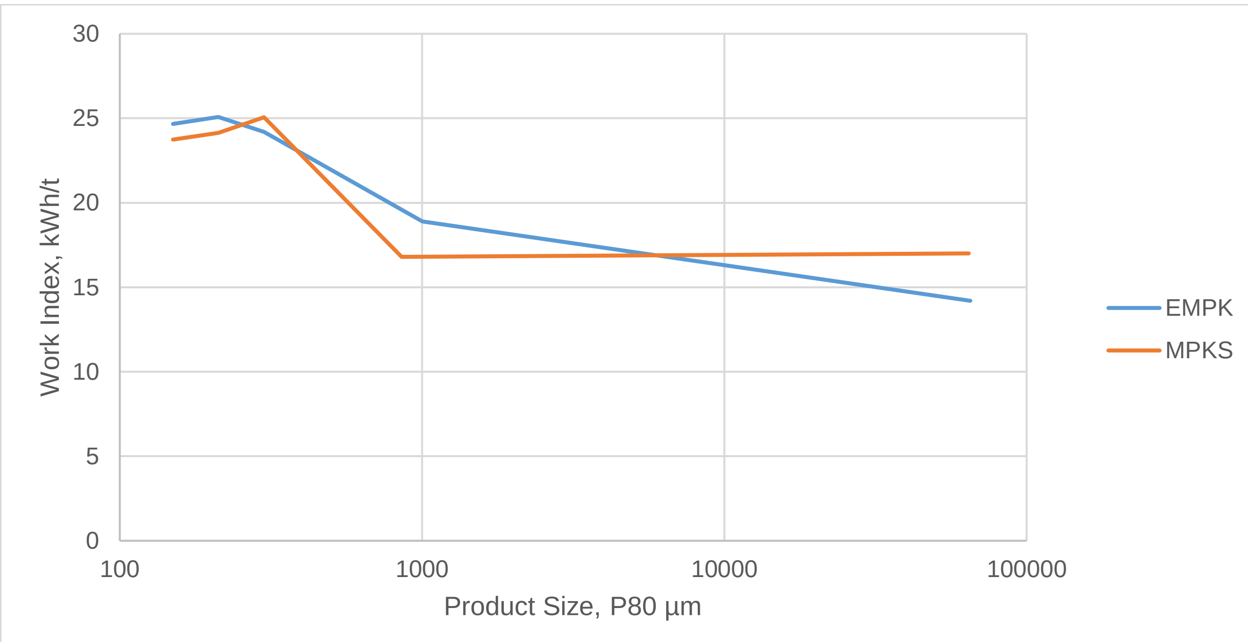
* Size Fraction Tested -31.5+26.5 mm
Source: BaseMet (2019)

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) to determine if the future material planned to be mined is different from the current material being treated in the AG Mill. A review of the samples tested demonstrated that there was not a significant difference between the pit bottom composite samples and the future drill core material. The samples tested are amenable to milling in the existing AG process plant.

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

Figure 13-4: Work Index versus Product Size



Source: JDS (2019)

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

- RWi vs. BWi demonstrated that the two samples are very similar and were amongst the hardest samples in the AGD global database. It was noted that historical results did not fit with the recent tests completed by BaseMet or the AGD global database;
- Drop Weight Axb vs. BWi showed minor differences between the drill core and bulk samples. The differences are due to apparatus and are therefore not significant. The BWi for the samples

indicated very hard material but the Axb 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 significant;
- BWi vs. Product Size P_{80} showed there was little variation in BWi kWh/t at the size fractions tested (300, 212, and 150). No significant difference was observed between the bulk and variability samples; and
- No significant difference between the bulk and variability samples was noted when comparing BWi in g/rev vs Product Size or Ore density vs. BWi in kWh/t.

13.4 Processing Assumptions

The current actual processing recoveries have been used within the mineral resource estimate to determine recoverable grades model curves for the Karowe ore.

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

14 Mineral Resource Estimate

The KDM has been in operation since 2012, and as of the end of June 2019, the mined open pit extends to a depth of approximately 156 m below surface. The 2019 mineral resource update for the KDM is predicated on the following information obtained since the previous mineral resource estimate completed in August 2018:

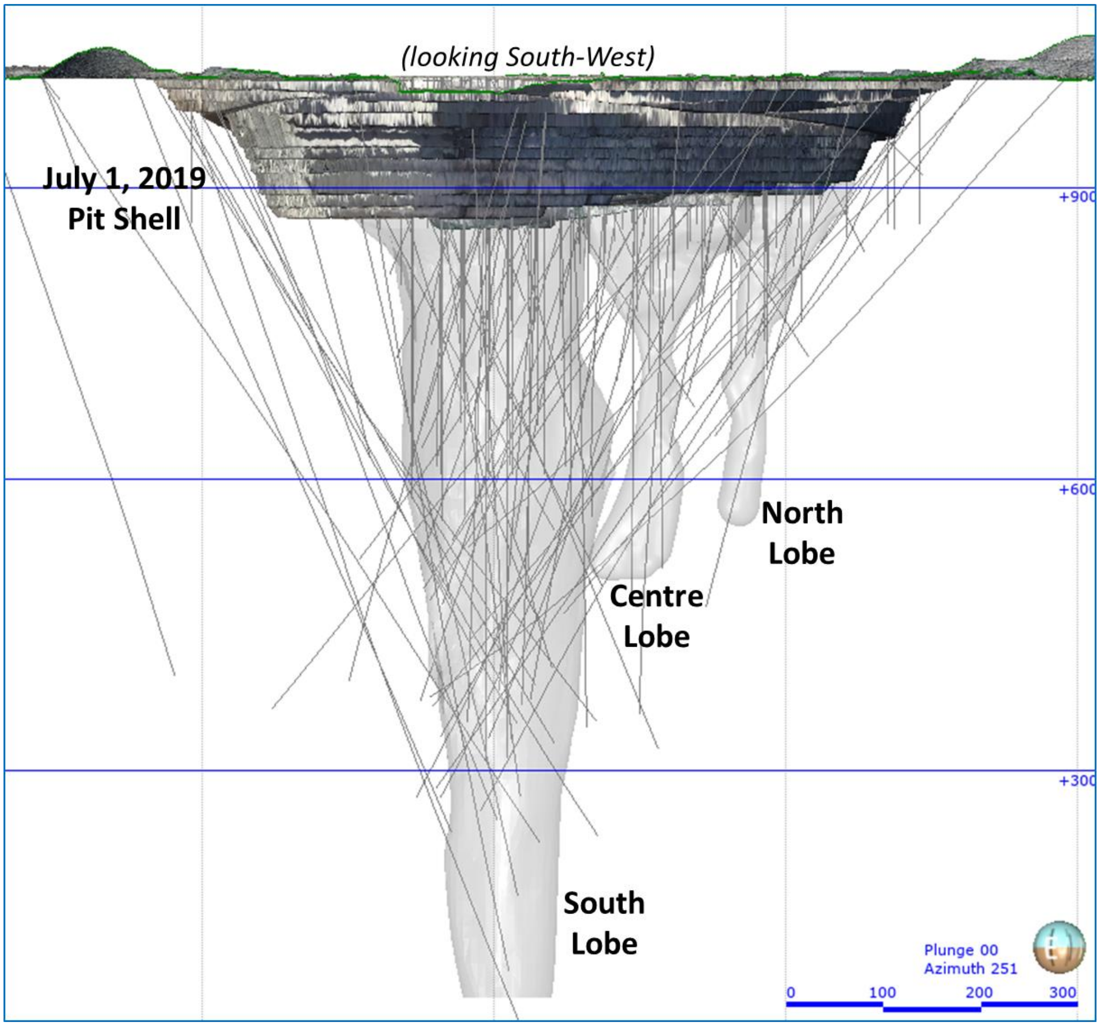
- Additional diamond core drilling conducted in 2018 and 2019 (located mainly below 600 masl within the South Lobe including a deep extension);
- An updated geological model for the South Lobe incorporating 2018 and 2019 drilling information;
- Additional microdiamond sampling of 2018 / 2019 drill holes (specifically targeting internal kimberlite domains within the South Lobe);
- In-pit mapping data of external kimberlite contacts within North, Centre and South Lobes;
- Updated Size Frequency Distributions (SFD) and revised diamond pricing information based on 2019 production and sales data; and
- As-built survey of the open pit mine as of July 1, 2019.

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

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

Figure 14-1 shows the geological model of the kimberlite, the mined open pit as of July 1, 2019, and all drilling used to support the 2019 Mineral Resource Estimate (MRE) for the KDM.

Figure 14-1: Geological Model of the Karowe Kimberlite



Note: Kimberlite pictured in (grey), the July 1, 2019 mined open-pit, and all drill hole traces
Source: SRK (2019)

The 2019 geological model update and MRE estimate were conducted in Seequent's Leapfrog Geo modeling software. The block model in comprised of a sub-block format using the following configuration parameters;

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

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

14.1 Resource Domains and Volumes

The internal geological model for Karowe is described in Section 7.3 of this report, and volume estimates of the unmined, in-situ internal kimberlite domains are listed in Table 14-1. All internal domains that have been mined as of July 1, 2019, are excluded from the volume estimates provided in Table 14-1.

Table 14-1: In-situ Volumes of Unmined Kimberlite Domains as of July 1, 2019

Kimberlite Domain	Volume (Million m^3)	Volume (% of total)
South_M/PK(S)	9.50	44.9%
South_EM/PK(S)	9.03	42.7%
South_KIMB3	0.32	1.5%
Centre	1.65	7.8%
North	0.65	3.1%
TOTAL	**21.13**	**100%**

Source: SRK (2019)

14.2 Bulk Density

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

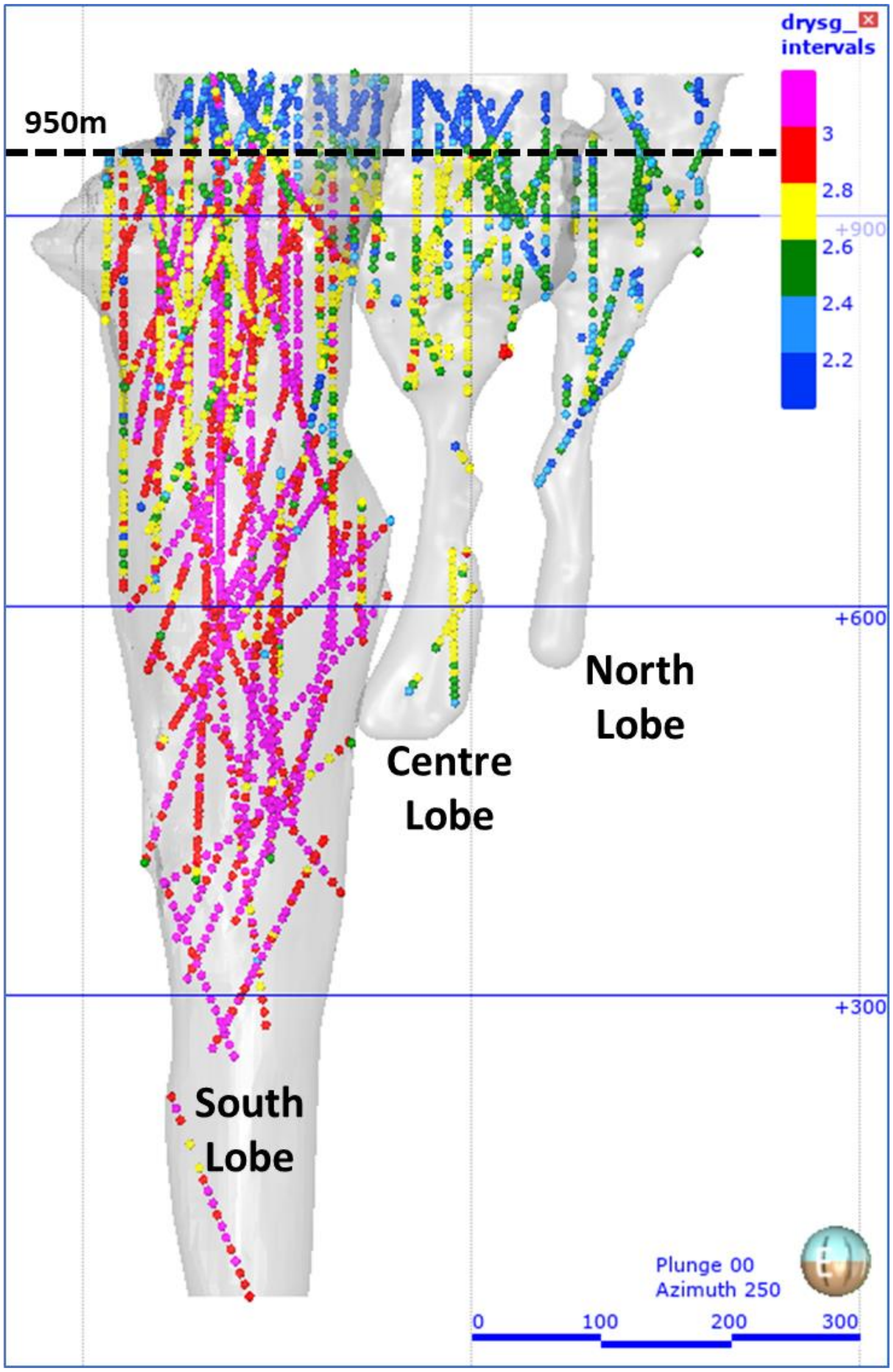
Additional dry density sample details for the two dominant kimberlite domains in the South Lobe (i.e. M/PK(S) and EM/PK(S)) are provided in Figure 14-3. As can be seen in the depth profiles for both the EM/PK(S) and M/PK(S) domains a relatively consistent dry density of 2.9 to 3.1 g/cm^3 is observed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

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

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

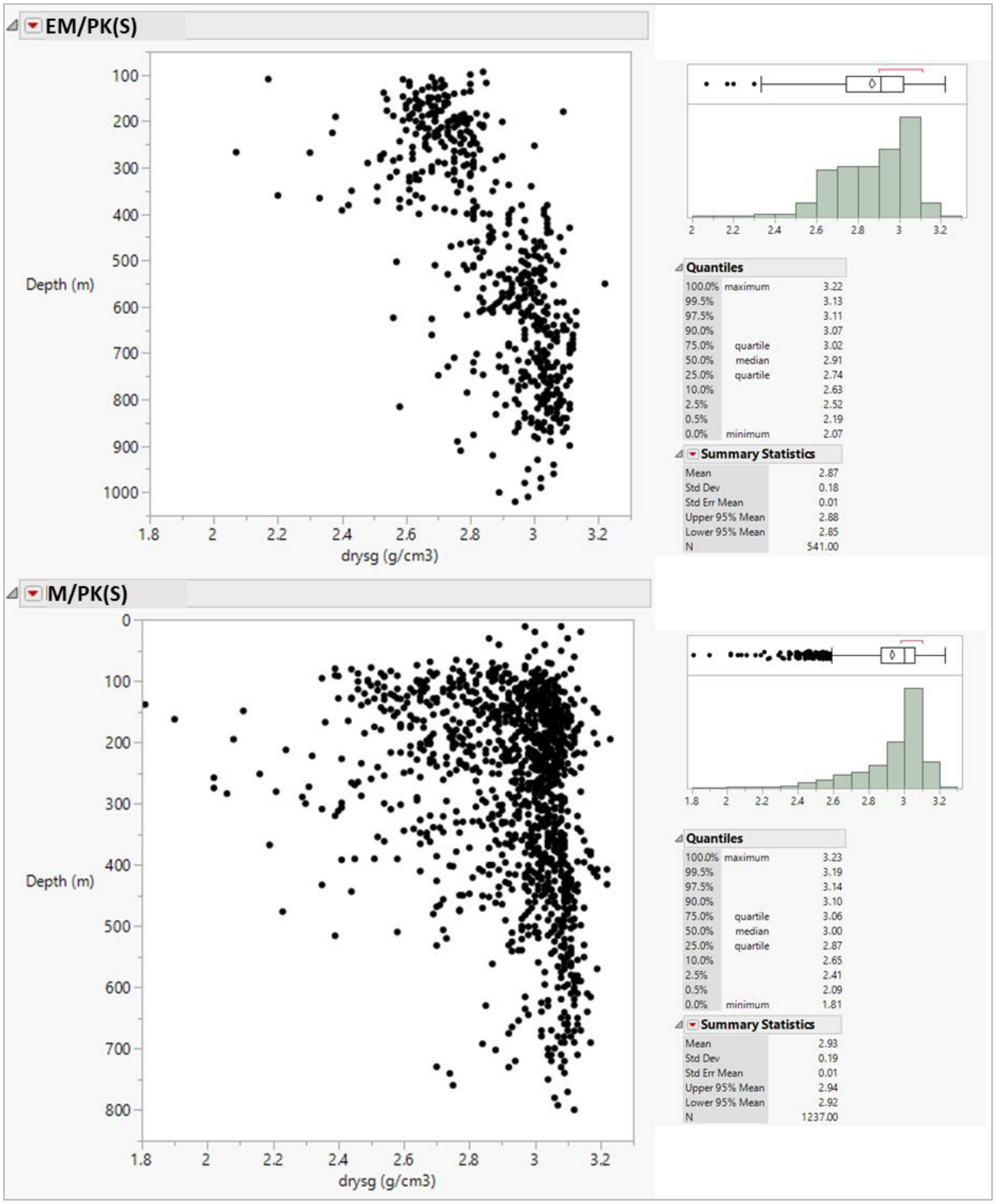
Note: (below 950 masl)
Source: SRK (2019)

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



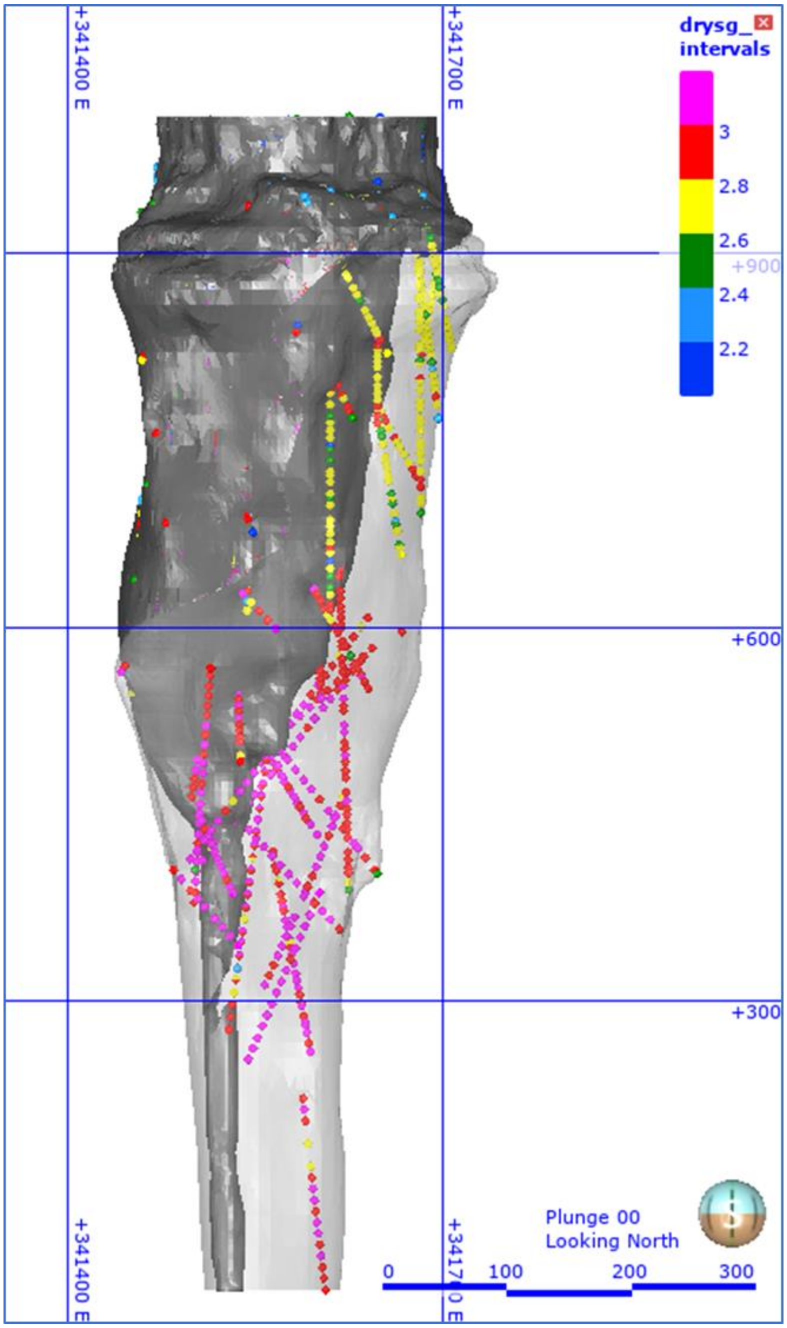
Note: (dry density units of g/m^3). Black dashed line at 950 masl demarcates approximate extent of upper weathered zone reflected in generally lower densities
Source: SRK (2019)

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



Source: SRK (2019)

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



Source: SRK (2019)

14.2.1 Bulk Density Estimation

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

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

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

Table 14-3: South Lobe Dry Density Variogram Parameters

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

Source: SRK (2019)

Table 14-4: Dry Density Estimation Parameters

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

Source: SRK (2019)

14.3 Grade Estimation

Diamond grade estimation has been conducted using two distinct methodologies:

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

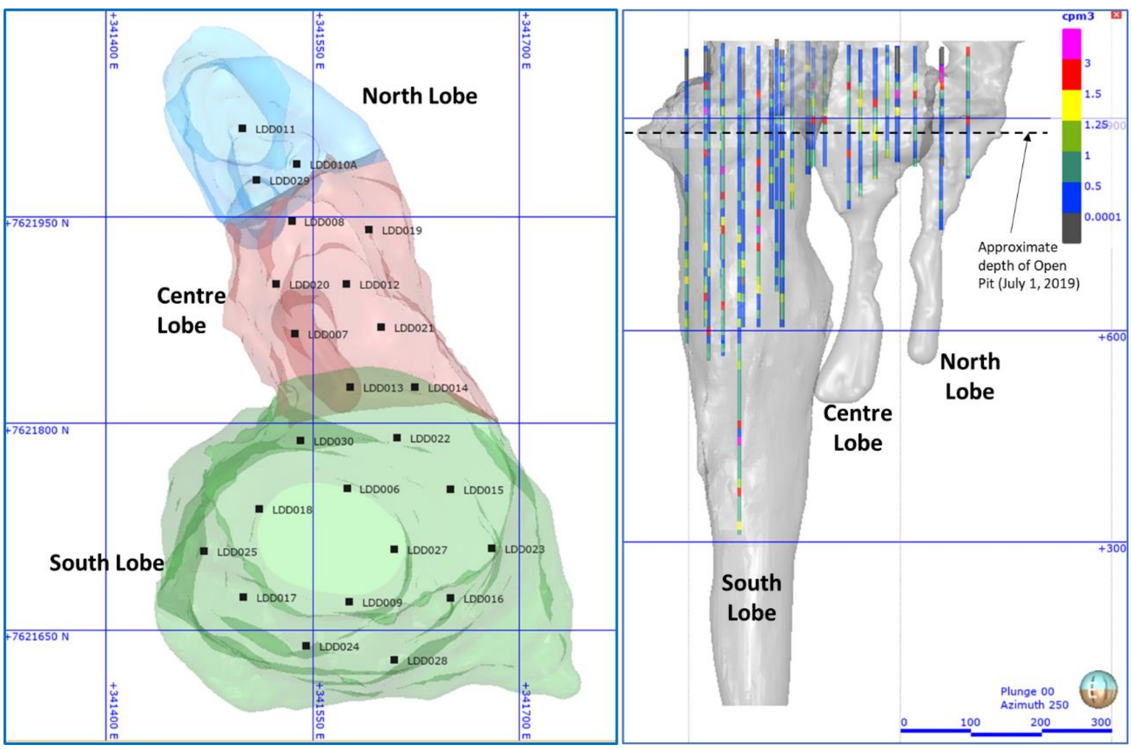
Global diamond grade estimation has solely been used within the deeper extents of South Lobe due to limited bulk sampling data available within this portion of the deposit.

14.3.1 Macrodiamond Data Summary

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

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

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



Note: Sample grades color-coded by diamond grade expressed in carats per m3 (cpm3)
Source: SRK (2019)

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

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

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

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

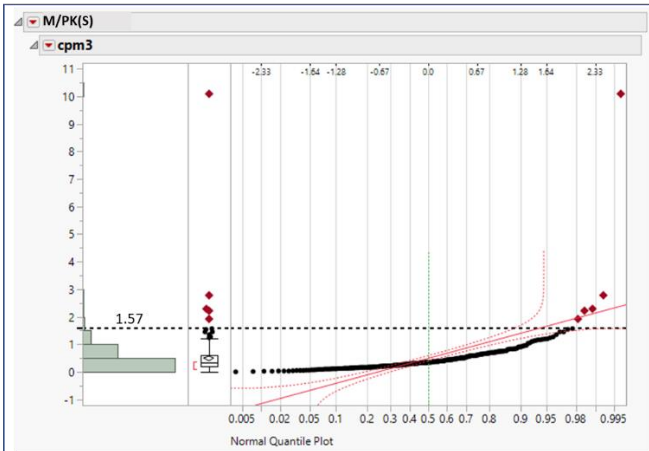
Source: SRK (2019)

14.3.2 Diamond Grade Capping Analysis

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

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

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



Uncapped Statistics

Quantiles

100.0%	maximum	10.10
99.5%		7.72
97.5%		1.55
90.0%		0.91
75.0%	quartile	0.61
50.0%	median	0.34
25.0%	quartile	0.19
10.0%		0.11
2.5%		0.03
0.5%		0.00
0.0%	minimum	0.00

Summary Statistics

Mean	0.49
Std Dev	0.71
Std Err Mean	0.04
Upper 95% Mean	0.58
Lower 95% Mean	0.40
N	264.00

Capped Statistics

Quantiles

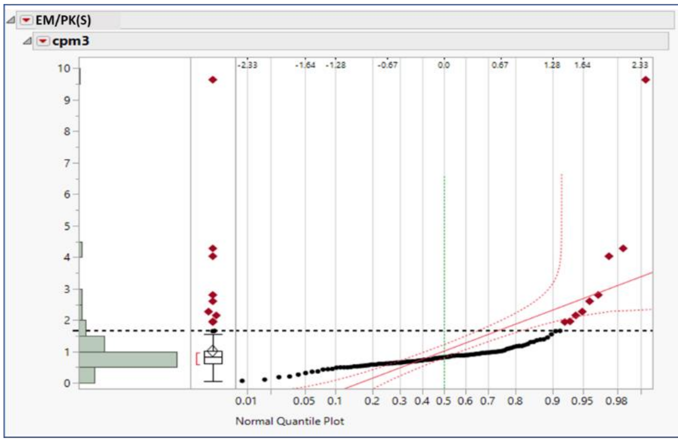
100.0%	maximum	1.57
99.5%		1.57
97.5%		1.55
90.0%		0.91
75.0%	quartile	0.61
50.0%	median	0.34
25.0%	quartile	0.19
10.0%		0.11
2.5%		0.03
0.5%		0.00
0.0%	minimum	0.00

Summary Statistics

Mean	0.45
Std Dev	0.35
Std Err Mean	0.02
Upper 95% Mean	0.49
Lower 95% Mean	0.40
N	264.00

Source: SRK (2019)

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



Uncapped Statistics

Quantiles

100.0%	maximum	9.64
99.5%		9.64
97.5%		4.04
90.0%		1.57
75.0%	quartile	1.02
50.0%	median	0.82
25.0%	quartile	0.62
10.0%		0.48
2.5%		0.19
0.5%		0.07
0.0%	minimum	0.07

Summary Statistics

Mean	1.01
Std Dev	1.01
Std Err Mean	0.09
Upper 95% Mean	1.20
Lower 95% Mean	0.83
N	117.00

Capped Statistics

Quantiles

100.0%	maximum	1.66
99.5%		1.66
97.5%		1.66
90.0%		1.57
75.0%	quartile	1.02
50.0%	median	0.82
25.0%	quartile	0.62
10.0%		0.48
2.5%		0.19
0.5%		0.07
0.0%	minimum	0.07

Summary Statistics

Mean	0.87
Std Dev	0.37
Std Err Mean	0.03
Upper 95% Mean	0.94
Lower 95% Mean	0.80
N	117.00

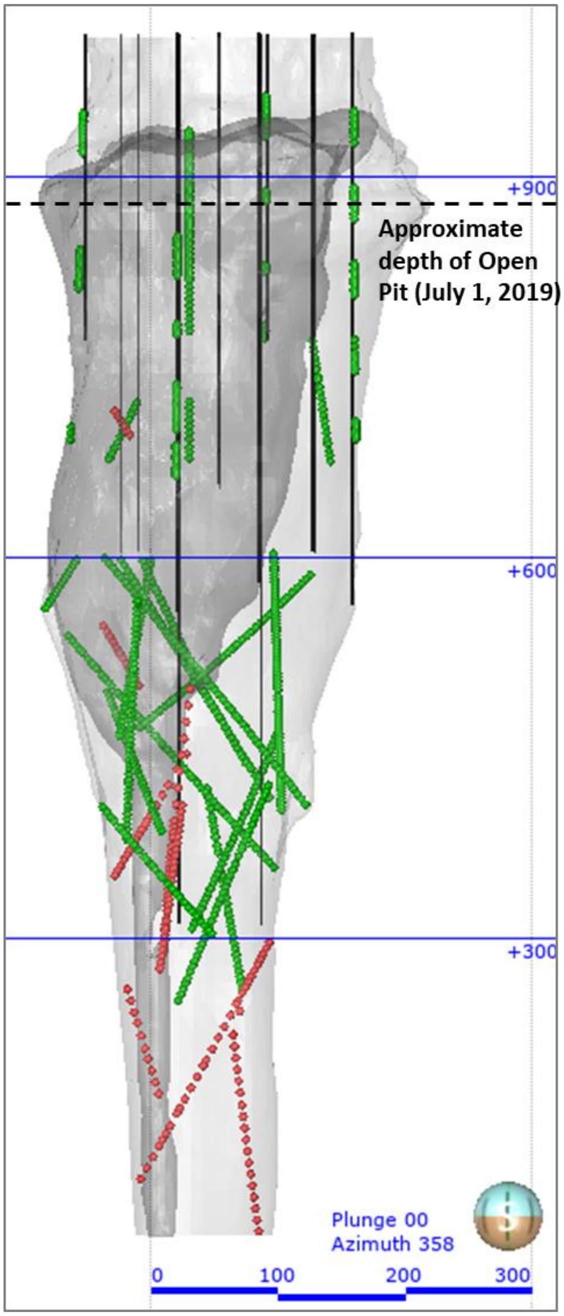
Source: SRK (2019)

14.3.3 Microdiamond Data Summary

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

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

Figure 14-8: Distribution of Microdiamond Samples



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

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

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

Source: SRK (2019)

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

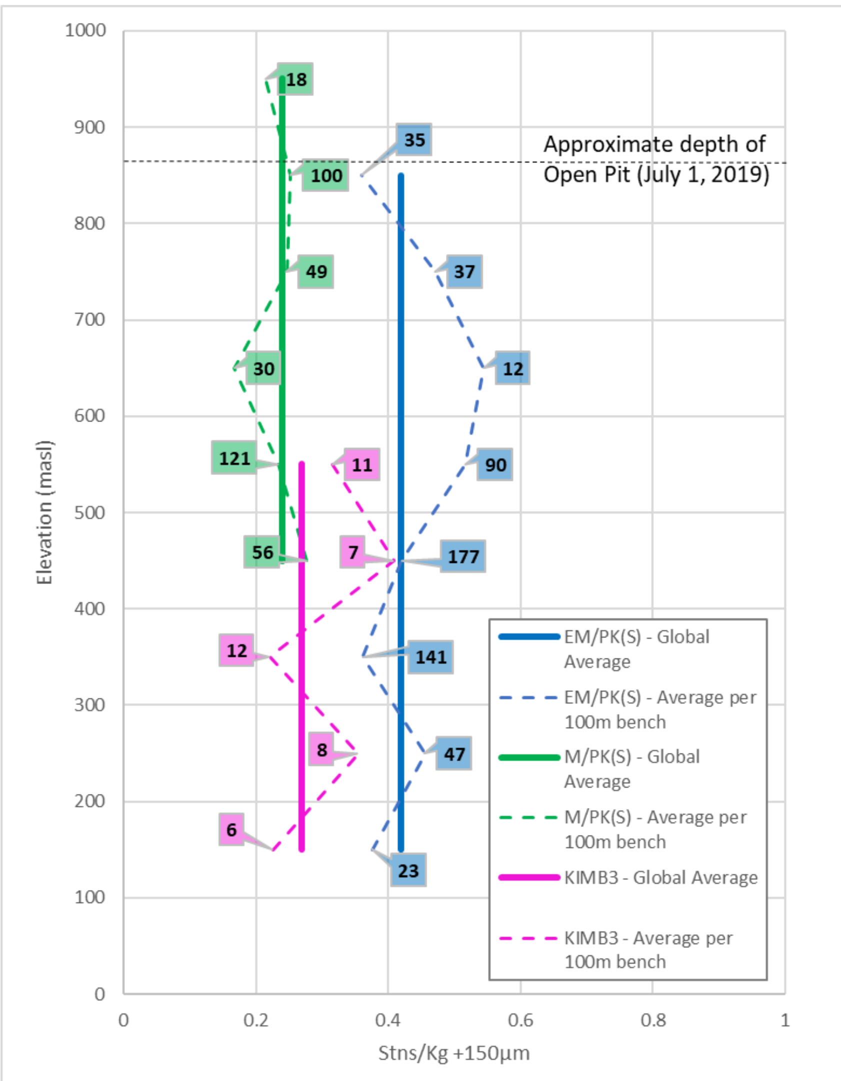
An SFD comparison for the EM/PK(S) 2017 and 2019 microdiamond populations is provided in Figure 14-10, which also demonstrates similar microdiamond population characteristics between the two sample groups. Therefore, no appreciable change in the microdiamond population within the EM/PK(S) domain occurs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

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

14-11 illustrates similar microdiamond size frequency distributions (SFDs) for the South EM/PK(S) and M/PK(S) domains, notwithstanding the noted differences in microdiamond and macrodiamond content.

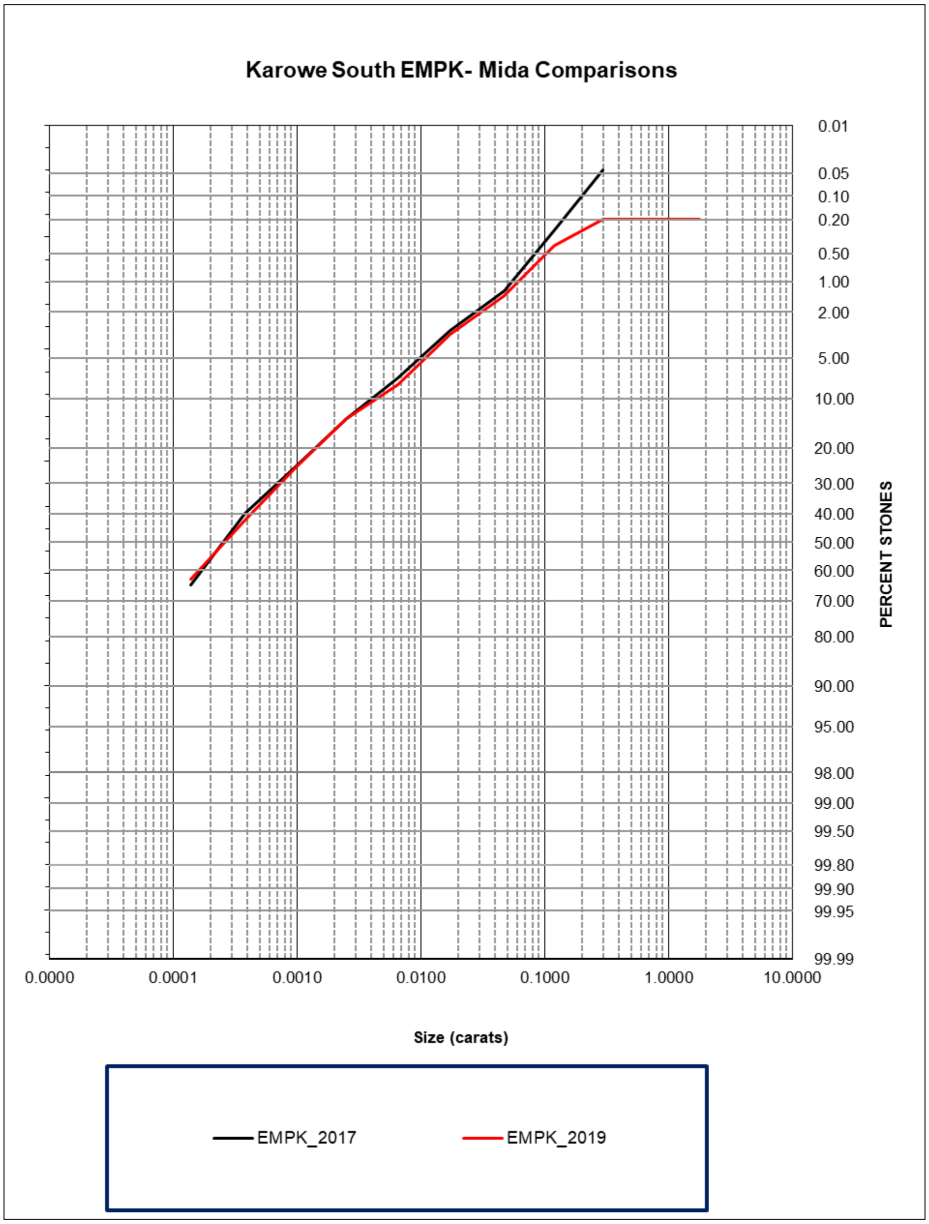
The limited microdiamond data obtained in 2019 for the KIMB3 domain provides a similar stone density to the M/PK(S) domain (Figure 14-9), however a finer SFD compared to both the South EM/PK(S) and M/PK(S) domains as depicted in Figure 14-11. As noted in Section 14.3.1, no bulk sampling of the KIMB3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

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



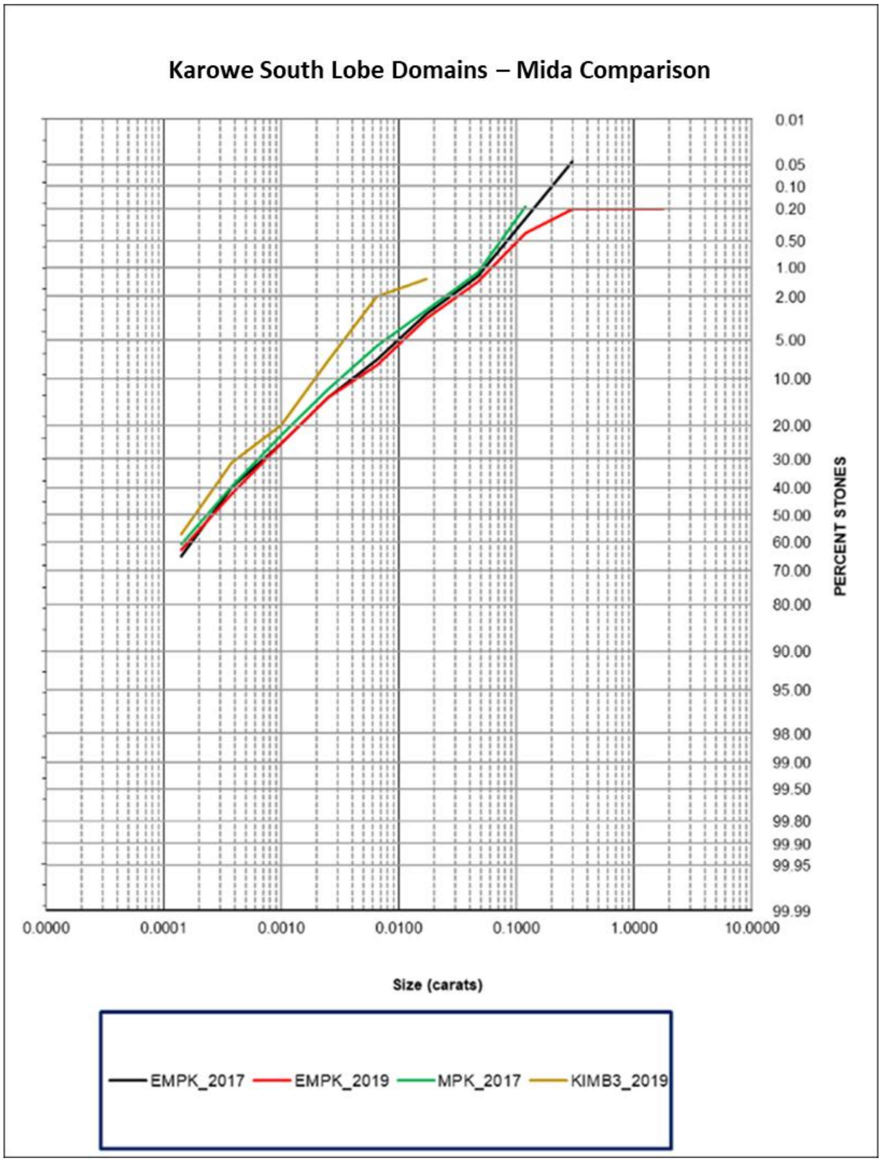
Note: (+150 µm) per 100 m vertical benches for South Lobe internal kimberlite domains. Global domain averages are provided as solid lines. Values in callout boxes represent the number of 8kg samples within each 100 m bench
Source: SRK (2019)

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



Source: SRK (2019)

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



Source: SRK (2019)

14.3.4 Local Grade Estimation

Similar to previous mineral resource estimates completed in 2009, 2014, 2017 and 2018, a local grade estimation approach has been utilized where spatially representative LDDH bulk sample data is available. However, the approach employed in 2019 has been modified to incorporate a hard boundary between the South Lobe M/PK(S) and EM/PK(S) domains due to the significant grade difference between these two domains. All previous mineral resource estimates disregarded the contact between the M/PK(S) and EM/PK(S) domains, and therefore a single diamond grade dataset was used for local block estimation within the South Lobe. The 2019 mineral resource estimate is comprised of local diamond grade estimates to the depth of LDDH bulk sampling within the South Lobe M/PK(S) and EM/PK(S) domains at 604 and 568 masl, respectively.

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

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

Table 14-7: South Lobe Diamond Grade Variogram Model

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

Source: SRK (2019)

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

Table 14-8: Diamond Grade Estimation Parameters

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

Source: SRK (2019)

14.3.5 Global Grade Estimation

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

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

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

- Microdiamonds from KIMB3 have a similar SFD as microdiamonds from the M/PK(S) domain (Figure 14-11). The ratio of micro- to macrodiamonds obtained for M/PK(S) material is hence assumed applicable to KIMB3; and
- A microdiamond stone density of 0.24 Stns/kg +150 µm for M/PK(S) correlates with a +1.0 mm macrodiamond content of 0.45 cpm^3.

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

14.3.6 Adjustment for Production Plant Recovery Efficiency

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

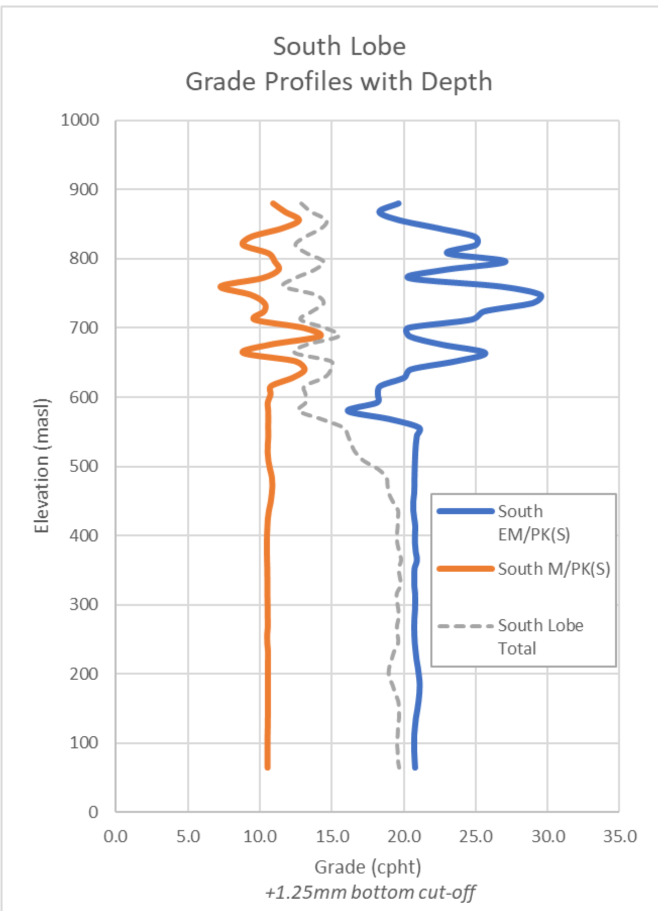
Over the course of 2018 and 2019, modifications within the Karowe process plant have improved the recovery efficiency of smaller diamonds within the mine production. Based on a comparison of quarterly mine production from Q4 2017 to Q3 2019, adjustment to the process recovery factor was required to reflect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

14.3.7 Grade Estimation Summary

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

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

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



Source: SRK (2019)

14.4 Diamond Value Estimate

Diamond value estimates presented in this section have been generated by Lucara and are based on LOM production and sales information to the end of August 2019. The diamond value estimates incorporate current trends observed through diamond tenders within 2019 and are representative of the current status of the diamond market. SRK has reviewed the information and analysis provided by Lucara and considers them to be reliable and consistent with average US$ per carat prices disclosed in Lucara quarterly financials.

Diamond value estimates are the product of the size frequency distribution of a given diamond population and the diamond quality characteristics of that population; and are typically unique for each kimberlite domain within a deposit. The 2019 mineral resource estimate for Karowe incorporates unique diamond value estimates for the two main kimberlite domains within the South Lobe (i.e. M/PK(S) and EM/PK(S)) based on discrete production and diamond sales data obtained from these domains. The North and Centre Lobe diamond value estimates remain unchanged since the previous mineral resource estimate completed in 2018.

14.4.1 Size Frequency Distribution Model

Details of the discrete production parcels used to develop SFD models for the North and Centre Lobes, and the South Lobe M/PK(S) and EM/PK(S) domains are provided in Table 14-9. Prior to 2019, a single diamond SFD model was used for the entirety of South Lobe because of limited discrete production data available for the EM/PK(S) domain due to its lack of exposure near surface. However, over the course of 2018 and 2019, mine production from the EM/PK(S) domain was possible allowing for the development of a distinct SFD model. It should be noted that for both the M/PK(S) and EM/PK(S) domains, the SFD models slightly underestimate the percentage of the +10.8 carat (ct) size class compared to the actual production parcels. This impact is discussed further in Section 14.4.2.

A comparison between the 2018 South Lobe SFD model and 2019 SFD models for the M/PK(S) and EM/PK(S) domains is provided in Table 14-10. The most significant change to note in these SFD models is within the +10.8 ct size fraction, which is associated with the most significant revenue component of the Karowe Mine production as further discussed in Section 14.4.2.

Table 14-9: Discrete Production Parcel Data for North Lobe, Centre Lobe, and South Lobe

Size Class	Discrete Production Parcels (cts per size class)				Discrete Production Parcel SFD's (% cts per size class)				2019 Model SFD's (% cts per size class)			
	M/PK(S)	EM/PK(S)	Centre	North	M/PK(S)	EM/PK(S)	Centre	North	M/PK(S)	EM/PK(S)	Centre	North
+10.8ct	25,802	3,933	8,836	579	6.3	8.3	3.4	1.0	5.9	8.0	3.1	1.0
6-10ct	11,852	1,417	5,626	1,140	2.9	3.0	2.2	2.0	3.5	3.6	2.9	2.4
3-5ct	23,854	2,739	14,378	3,552	5.8	5.8	5.6	6.2	5.8	5.6	3.9	5.3
8-10gr	22,166	2,156	14,263	4,058	5.4	4.6	5.5	7.1	4.5	4.1	7.2	7.7
3-6gr	71,559	6,410	50,292	14,732	17.5	13.6	19.6	25.7	18.2	14.0	19.4	25.7
+11DTC	75,466	7,695	53,852	14,130	18.4	16.3	20.9	24.7	18.4	16.3	21.0	24.7
+9DTC	62,232	6,763	41,516	9,116	15.2	14.4	16.1	15.9	15.2	14.4	15.9	15.9
+7DTC	46,027	5,150	28,524	5,288	11.2	10.9	11.1	9.2	11.2	10.9	11.0	9.2
+5DTC	62,701	8,892	36,214	4,584	15.3	18.9	14.1	8.0	15.3	18.9	14.0	8.0
+3DTC	7,985	1,949	3,686	73	1.9	4.1	1.4	0.1	2.0	4.1	1.3	0.1
Total Carats	**409,644**	**47,103**	**257,187**	**57,252**								

Note: Size class abbreviations are “DTC” = Diamond Trading Company, “gr” = grainer, and “ct” = carats and resultant SFD models at +1.25mm bottom cut-off.
Source: SRK (2019)

Table 14-10: Comparison of 2018 and 2019 SFD Models for South Lobe

Size Class	SFD Models (% cts per size class)		
	South 2018	M/PK(S) 2019	EM/PK(S) 2019
+10.8 ct	6.4	5.9	8.0
6-10 ct	4.4	3.5	3.6
3-5 ct	5.9	5.8	5.6
8-10 gr	5.3	4.5	4.1
3-6 gr	17.0	18.2	14.0
+11 DTC	18.2	18.4	16.3
+9 DTC	15.3	15.2	14.4
+7 DTC	10.7	11.2	10.9
+5 DTC	15.1	15.3	18.9
+3 DTC	1.7	2.0	4.1

Source: SRK (2019)

14.4.2 Value Distribution Models

The 2019 value distribution models are provided in Table 14-11, and are based on discrete mine production data for each kimberlite domain obtained since the start of mining and diamond sales information to the end of August 2019. The average US$/ct estimate for the North and Centre Lobes are unchanged from 2018, however the South Lobe M/PK(S) and EM/PK(S) domains now reflect unique US$/ct estimates based on the individual SFD models discussed in Section 14.4.1. As shown in Table 14-11, the average value per size class for the M/PK(S) and EM/PK(S) domains are very similar and reflect similar diamond quality characteristics between these two domains. However, the overall higher average US$/ct for the EM/PK(S) domain reflects the coarser diamond SFD for this domain specifically within the +10.8 ct size fraction.

Table 14-11: 2019 Value Distribution Models for Karowe

Size Class	2019 Model SFD's (% cts per size class)				2019 Value per Size Class (US$/ct)				2019 Revenue per size class (US$/ct) Model SFD's			
	North	Centre	M/PK(S)	EM/PK(S)	North	Centre	M/PK(S)	EM/PK(S)	North	Centre	M/PK(S)	EM/PK(S)
+10.8 ct	1.0	3.1	5.9	8.0	1,600	6,225	7,600	7,600	15	190	449	606
6-10 ct	2.4	2.9	3.5	3.6	1,127	1,194	1,108	1,112	27	35	39	41
3-5 ct	5.3	3.9	5.8	5.6	808	669	680	682	43	26	39	38
8-10 gr	7.7	7.2	4.5	4.1	484	435	446	446	37	31	20	18
3-6 gr	25.7	19.4	18.2	14.0	223	209	224	222	57	41	41	31
+11 DTC	24.7	21.0	18.4	16.3	95	95	102	102	23	20	19	17
+9 DTC	15.9	15.9	15.2	14.4	64	65	72	72	10	10	11	10
+7 DTC	9.2	11.0	11.2	10.9	56	56	51	51	5	6	6	6
+5 DTC	8.0	14.0	15.3	18.9	47	48	43	43	4	7	7	8
+3 DTC	0.1	1.3	2.0	4.1	34	42	39	39	0	1	1	2
							Average US$/ct		**222**	**367**	**631**	**777**

Source: SRK (2019)

As mentioned in Section 14.4.1, the modeled SFD's for the South Lobe M/PK(S) and EM/PK(S) domains slightly underestimate the proportion of +10.8 ct diamonds when compared to the actual production diamond SFD's as shown in Table 14-9. The impact on the *average* US$/ct for the M/PK(S) and EM/PK(S) domains is a reduction of US$24/ct and US$23/ct, respectively, compared against actual production. Diamond prices used in the 2019 mineral resource estimate accordingly reflect a conservative value model compared to actual production.

Value models exclude from the pricing approximately US$250 M in revenue generated from +US$10 M single stones (i.e. exceptional stones) sold since 2014, which includes the Constellation diamond (813 ct sold for US$63 M) and the Lesedi la Rona diamond (1,109 ct sold for US$53 M). Revenues from the sale of such exceptional diamonds vary materially through time, though represent approximately 15.6 percent of all diamond sales revenue since the start of commercial production in April 2012. Total sales of approximately 2.8 M carats since the start of commercial production have generated revenue of US$1.6 B, for a LOM average price per carat of US$586/ct (including exceptional stones). Excluding revenues from both the Constellation and Lesedi La Rona diamonds, the LOM average price per carat is US$509/ct.

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

14.5 Mineral Resource Statement and Classification

A mineral resource is defined by the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) as;

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

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

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

The 2019 mineral resources for the KDM have been classified as either Indicated or Inferred Mineral Resources. No Measured Mineral Resource has been defined for this deposit. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) define Indicated and Inferred Mineral Resources as follows;

Indicated Mineral Resource

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

Inferred Mineral Resource

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

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

The 2019 Mineral Resource statement for the Karowe Diamond Mine is provided in Table 14-12, which is inclusive of Mineral Reserves.

Table 14-12: Karowe Diamond Mine 2019 Mineral Resource Statement

Classification	Domain	Volume (Mm^3)	Tonnes (Mt)	Density (t/m3)	Carats (Mcts)	Grade (cpht)	Average US$/ct
Indicated	South_M/PK(S)	9.40	27.81	2.96	3.01	10.8	631
	South_EM/PK(S)	7.62	22.10	2.90	4.68	21.2	777
	Centre	1.28	3.28	2.57	0.50	15.1	367
	North	0.44	1.08	2.45	0.13	11.8	222
TOTAL INDICATED		**18.74**	**54.27**	**2.90**	**8.32**	**15.3**	**690**
Inferred	South_M/PK(S)	0.10	0.31	3.05	0.03	10.5	631
	South_EM/PK(S)	1.40	4.18	2.97	0.87	20.9	777
	South_KIMB3	0.32	0.94	2.94	0.10	10.9	631
TOTAL INFERRED		**1.82**	**5.42**	**2.97**	**1.01**	**18.6**	**750**

Notes:
1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All numbers have been rounded to reflect accuracy of the estimate.
2. Mineral Resources are in-situ Mineral Resources and are inclusive of in-situ Mineral Reserves.
3. Mineral Resources are exclusive of all mine stockpile material.
4. Mineral Resources are quoted above a +1.25 mm bottom cut-off and have been factored to account for diamond losses within the smaller sieve classes expected within a commercial process plant.
5. Inferred Mineral Resources are estimated on the basis of limited geological evidence and sampling, sufficient to imply but not verify geological grade and continuity. They have a lower level of confidence than that applied to an Indicated Mineral Resource and cannot be directly converted into a Mineral Reserve.
6. Average diamond value estimates are based on 2019 diamond sales data provided by Lucara Diamond Corp.
7. Mineral Resources have been estimated with no allowance for mining dilution and mining recovery.

(effective date of July 1, 2019)
Source: SRK (2019)

14.6 Previous Mineral Resource Statement

The previous mineral resource estimate for the KDM reflects mine depletion up to December 31, 2017 and is provided in Table 14-13. The previous Mineral Resource was quoted using a bottom cut-off of +1.25 mm, based on a process recovery factor attributable to the Karowe process plant configuration at that time. The average US$/ct value quoted was based on historical production and sales data incorporating the first 3 months of 2018.

Table 14-13: Karowe 2018 Mineral Resource Statement (effective date December 31, 2017)

Classification	Kimberlite Lobe	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	Carats (Mct)	Grade (cpht)	Average US$/ct
Indicated	**South Lobe**	16.29	2.92	47.63	6.78	14.2	716
	Centre Lobe	1.68	2.57	4.32	0.63	14.6	367
	North Lobe	0.62	2.48	1.54	0.20	13.0	222
TOTAL INDICATED		**18.59**	**2.88**	**53.48**	**7.62**	**14.2**	**674**
Total Inferred	**South Lobe**	**1.93**	**3.02**	**5.84**	**1.17**	**20.0**	**716**

Source: Mineral Services (2018)

14.7 Recommendations

The following recommendations are provided to continue to advance the understanding of the Mineral Resource for the Karowe Diamond Mine:

- Further drilling and sampling (microdiamond and/or bulk sampling) is required to upgrade Inferred Mineral Resources to higher confidence levels;
- Additional drilling and geological assessment is required to determine the impact of localized variants of the main kimberlite units encountered within the South Lobe.
- Additional drilling is required to confirm the modelled but not drill-confirmed extent of the M/PK(S) domain below 438 masl elevation;
- Further spatial correlation of large diamond recoveries from production relative to LDDH bulk sample data should be undertaken to determine if enhanced large stone predictive capabilities could be established;
- Continued incorporation of pit geological mapping is recommended to enhance internal kimberlite domain definition;
- Continued reconciliation of production forecasts relative to mine production is recommended to assess the robustness of mineral resource estimates; and
- Continued refinement of kimberlite domain SFDs based on additional discrete production data is also recommended.

15 Mineral Reserve Estimate

15.1 Open Pit

Open pit reserves have been provided to JDS for inclusion of the FS. Open pit reserves have been validated by JDS.

The mineral resource estimate and block model was updated in late September 2019. The open pit mine production schedule corresponds to the LOM schedule and end of period maps prepared by Lucara in September 2019, using the previous mineral resource estimate. The LOM end of period maps were used to update the production schedule and mineral reserve with the 2019 mineral resource estimate update. The open pit design and mining schedule has not been optimized based on the 2019 mineral resource estimate update but will be a focus for work starting in December 2019. Further work is not expected to materially change the mineral reserve estimate.

15.2 Underground

Underground mine reserves were prepared by Gord Doerksen, P.Eng. of JDS and include the fully diluted and recovered mineable resources below the open pit.

15.2.1 Underground Cut-off Grade Criteria

Underground mining reserve estimates were calculated from resource block model tonnes and grades to define a diamond cut-off grade (COG) to determine the mineable portions of the South Lobe. The mineable resource was defined based on COG values greater than 5.51 cpht after dilution and mining recoveries are applied. All of the kimberlite material in the South Lobe is above the cut-off value.

Cut-off grade parameters include diamond valuation, payable content, royalties, corporate costs and sales charges, and estimated operating costs, dilution, and recoveries.

Diamond valuation was derived from historical sales adjusted for current and estimated future values and weighted against resource lithologies to arrive at an average cost per carat. Off-site, in-country corporate costs such as Lucara Botswana management, cost of sales, and costs associated with Clara have been provided by Lucara and are included as Sales and Corporate Costs in the cut-off grade calculation. Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the mineral resource block model assumptions and therefore have taken recoveries into account. Operating costs were derived from existing operational charges, previous studies, and benchmarking local mines.

Parameters used for cut-off grade calculations may not reflect exact parameters used for the economic model as several items were not yet refined at the time of preparation.

The cut-off grade parameters are shown in Table 15-1.

Table 15-1: Underground Cut-Off Grade Parameters

Parameter	Unit	Value
Revenue, smelting & refining		
Diamond Price	US$/ct	681.00
Payable content	%	100%
Royalty (10%)	US$/ct	68.10
Sales & Corporate Costs	US$/ct	31.00
Diamond value per carat	US$/ct	581.90
Operating Costs		
Mining	US$/t milled	9.00
Processing	US$/t milled	16.00
G&A	US$/t milled	6.00
Total OPEX estimate	US$/t milled	31.00
Mining Recovery and Dilution		
Mining Recovery	%	100.0
Mining Dilution	%	3.5
Cut-off Grade	**cpht**	**5.51**

Source: JDS (2019)

15.2.2 Underground Dilution

A total dilution of 3.5% has been included in the underground reserve estimate. Three types of underground dilution were applied to the stope and development designs:

- External Dilution;
- Internal Dilution; and
- Inferred Dilution.

15.2.2.1 External Dilution

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

The large, continuous nature of the resource combined with excellent ground conditions in both the kimberlite and most of the host rock suggests little to no dilution will occur in the granite lithology domains. Above the granite, a five percent overbreak / slough dilution has been included to resources within 15 m of the circumference of the South Lobe, as well as the crown pillar separating the underground from open pit. External dilution comprises approximately 569 kt or 1.7% of the reserve.

15.2.2.2 Internal Dilution

Internal dilution, or designed dilution, accounts for additional, lower than COG material within the planned stope or development design shape. Grades for internal dilution are taken from the mineral resource model if available. The resource, albeit relatively uniform, undulates along the contact between the kimberlite and host rock. As such, drill and blast practices will naturally include some wall rock within the stope design. Internal dilution comprises approximately 258 kt or 0.8% of the reserve.

15.2.2.3 Inferred Resource Dilution

Any Inferred Resource class material within the mining reserve stope and development shapes has been treated as waste and has been assigned zero value. Inferred dilution comprises approximately 317 kt or 1.0% of the reserve.

15.2.3 Mining Recovery

A 100% mine recovery has been assumed for the reserves. Process recovery has been included within the resource block model estimation and as such, is not required in the cut-off grade estimation.

15.3 Mineral Reserve Estimate

The effective date for the Mineral Reserve Estimate is September 26, 2019 and the estimate was prepared by QP Gord Doerksen, P.Eng. All Mineral Reserves in Table 15-2 are classified as Probable Mineral Reserves. The Mineral Reserves, except stockpiles, are not in addition to the Mineral Resources, but are a subset thereof.

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

Table 15-2: Karowe Mine Mineral Reserve Estimate

Lobe - Type	Classification	Ore (Mt)	Diluted Grade (cpht)	Contained Carats ('000s ct)	Price (US$/ct)
Open Pit					
North	Probable	0.6	10.0	56	222
Centre	Probable	3.2	15.1	478	349
South – EM/PK(S)	Probable	3.6	23.9	850	777
South – M/PK(S)	Probable	10.2	10.8	1,098	631
Open Pit	**Total**	**17.4**	**14.2**	**2,481**	**618**
Underground					
South – EM/PK(S)	Probable	16.3	19.9	3,246	777
South – M/PK(S)	Probable	17.1	10.6	1,807	631
Underground	**Total**	**33.5**	**15.1**	**5,053**	**725**
Stockpiles					
North	Probable	0.4	12.7	51	222

Lobe - Type	Classification	Ore (Mt)	Diluted Grade (cpht)	Contained Carats ('000s ct)	Price (US$/ct)
Centre	Probable	0.4	12.8	54	349
South – M/PK(S)	Probable	1.6	9.5	151	631
Mixed	Probable	4.0	5.0	198	609
Stockpiles	**Total**	**6.4**	**7.1**	**454**	**542**
Combined					
All	**Total**	**57.3**	**13.9**	**7,988**	**681**

1. Prepared by Gord Doerksen, P.Eng. JDS Energy & Mining Inc.
2. CIM definitions were followed for Mineral Reserves and the effective date of the Mineral Reserve is September 26, 2019.
3. Mineral Reserves are estimated based on an UG mining cost of US$9/t, a processing cost of US$16/t and a G&A cost of US$6/t. Process recovery of the diamonds was assumed to be 100% as the recoveries were included in the mineral resource block model assumptions and therefore have taken recoveries into account. All of the kimberlite material in the South Lobe is above the cut-off value.
4. Diamond valuation was derived from historical sales adjusted for current and estimated future values.
5. Tonnages are rounded to the nearest 100,000 tonnes; diamond grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained diamonds are reported as thousands of carats.

Source: JDS (2019)

16 Mining Methods

16.1 Introduction

KDM is an existing open pit mine located in Central Botswana that has been in production since 2012 and has extracted approximately 20 Mt of ore to date. Conventional open pit drill and blast mining with diesel excavators and trucks provide an average annual 2.6 Mt of kimberlite feed to the mill, plus additional ore to surface stockpiles. The open pit mine operation is expected to terminate mid-2025, ending at an elevation of 710 masl. The mine currently has approximately two years of stockpiled reserves available for processing.

There are substantial resources remaining below the economic extents of the open pit that may be extracted by underground mining methods. This opportunity was initially evaluated through a preliminary economic analysis (PEA) completed by Royal Haskoning DHV (RH) in November 2017 (Oberholzer, 2017). This PEA considered block caving (BC), sub level caving (SLC), and longhole open stoping (LHOS) mining methods. SLC with ramp access was recommended due to superior economics, however, geotechnical risks were identified with ramp advancement through stratigraphic units of weaker ground. The PEA identified the need for more detailed trade-off studies to select the appropriate means of underground access and mine method. As a result, in 2018 Lucara Diamonds elected to conduct an internal study to further investigate the mining approach recommended in the PEA, and subsequently commissioned JDS in 2019 to prepare a FS on KDM and re-evaluate the optimal mine method and means of access for the deposit.

This FS investigated several underground mining methods based on data and information from an exhaustive field program conducted in 2018 and 2019 to define mineral resource, geotechnical, and hydrogeological characteristics necessary for making informed decisions at a FS-level study. The mining methods considered in the PEA were included as well as the addition of pre-conditioned block caving and long hole shrinkage (LHS). The small hydraulic radius at depth (27 m), low in-situ (horizontal) stress, and high compressive strength of the kimberlite suggested that the resource will not cave with or without pre-conditioning and will therefore require drill and blast assistance, leaving SLC, LHOS, and LHS as options. The mine plan favours LHS over these three options from both an economic, practicality, and risk mitigation standpoint and LHS was ultimately selected for this FS.

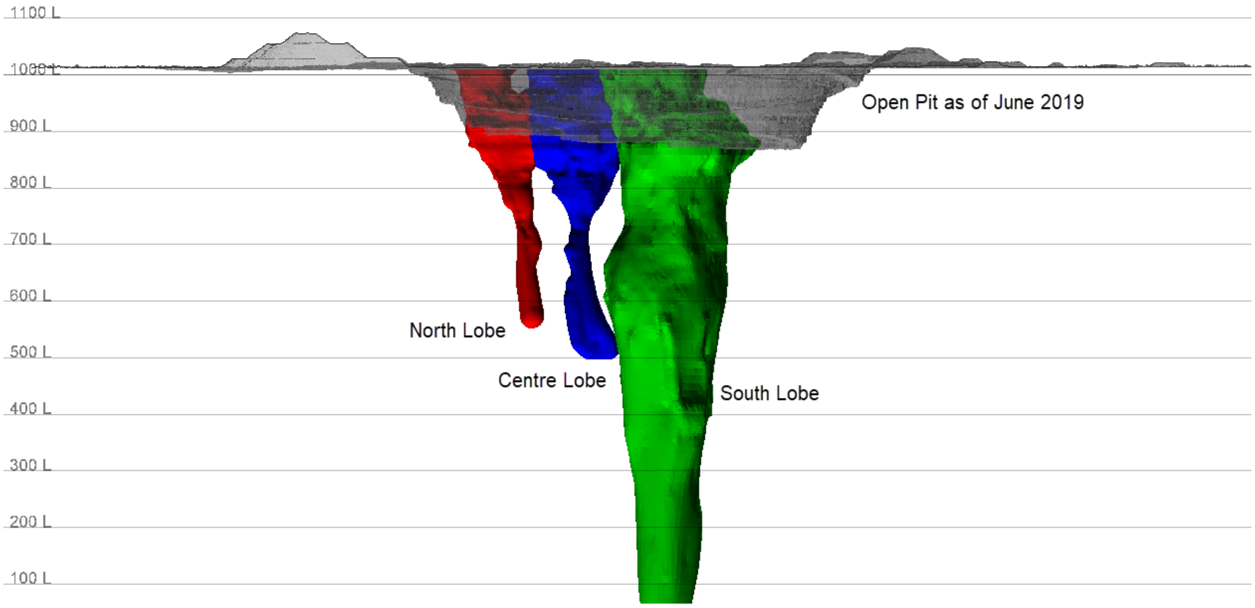
The mine design and planning for KDM is based on the resource model completed by SRK in 2019, as detailed in Section 14 of this report. The mine plan proposes the continuation of open pit activities to a depth of 710 masl at which point the resource is to be mined by underground methods to a depth of 310 masl. The mine will provide on average 2.6 Mt/a to the processing facility and add 13 years to the mine life. The mine method and production schedule has been selected to provide uninterrupted mill feed during the transition from open pit to underground operations. A total of 33.5 Mt with an average grade of 15.1 cpht will be mined from the underground operations. Underground development will begin in 2020 with full production ramp up completing in 2025. Stockpiles will be available on surface should they be needed during the OP to UG transition.

16.2 Deposit Characteristics

The Karowe resource contains three distinct coalescing pipes, referred to as the North, Centre, and South Lobes as illustrated in Figure 16-1. All lobes are outcropping, dip vertically, and vary in diameter and depth.

The South Lobe is the largest of the three, and its Indicated Resources extend approximately 760 metres below surface (from 1,010 masl to 250 masl). The North and Centre lobes extend below the open pit limit but have been excluded from the planned underground mine as they are inferred at depth and are of low value.

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



Source: JDS (2019)

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

Table 16-1: South Lobe Dimensions and Hydraulic Radius

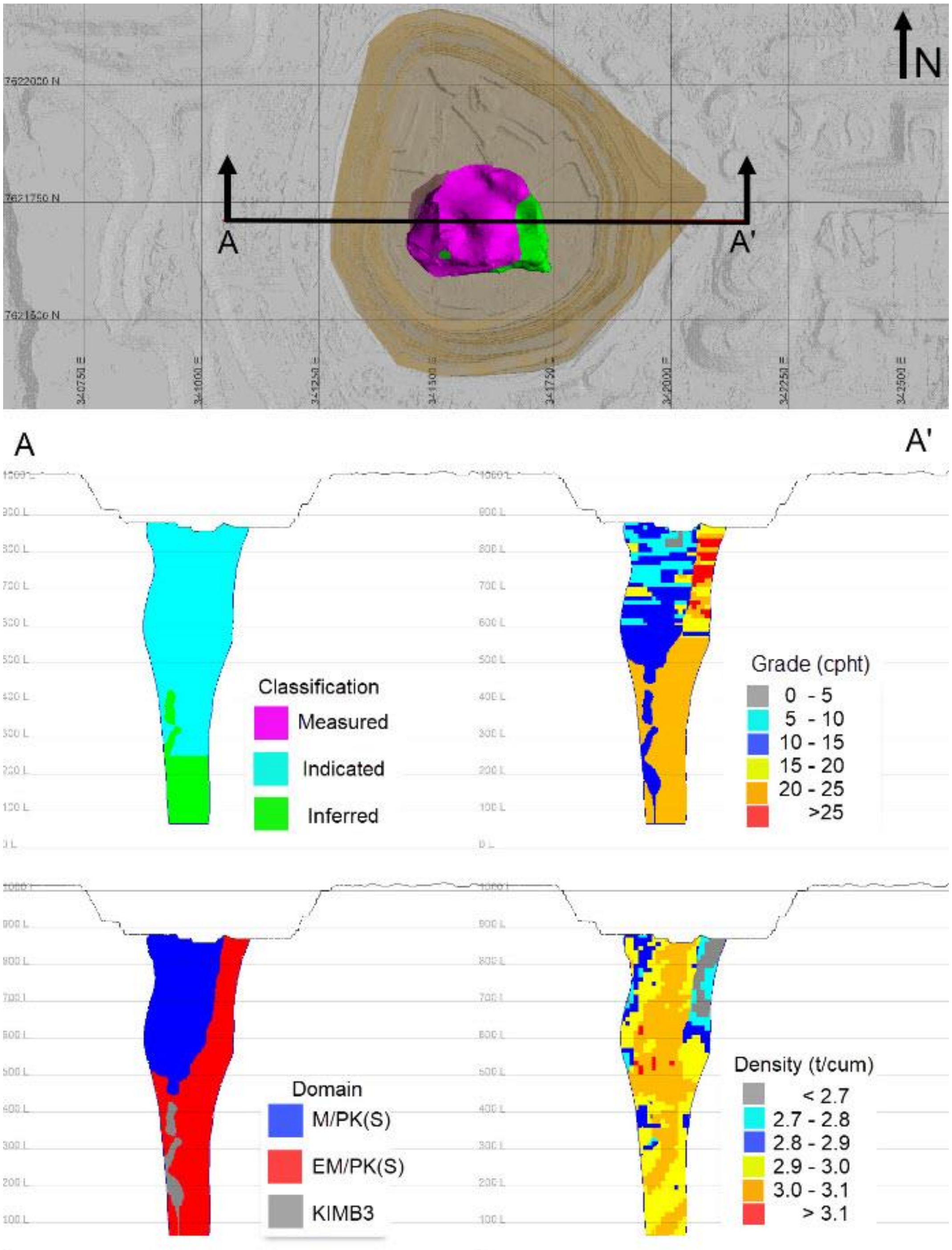
Elevation (masl)	Diameter (m)	Area (m^2)	Circumference (m)	Hydraulic Radius
800	215	36,400	703	52
700	207	33,550	668	50
600	213	35,575	704	51
500	180	25,330	592	43
400	152	18,130	528	34
300	122	11,680	389	30
200	110	9,560	355	27
100	101	8,060	325	25

Source: JDS (2019)

The South Lobe contains four distinct domains, each with unique mineral properties. These domains are discussed in greater detail in Chapter 6 and are summarized as EM/PK(S), M/PK(S), KIMB3, and

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

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



Source: JDS (2019)

16.3 Geotechnical Analysis and Recommendations

16.3.1 Introduction

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

16.3.2 Geotechnical Data Collection

A geotechnical investigation program was carried out to support underground mine design, building on the open pit and underground PEA geotechnical modelling carried out in 2017. The geotechnical drilling, sampling and testing program was designed to comply with the data confidence requirements of a FS, in support of a feasibility-level mine design, and leading into optimization of the design implementation. The investigation focused on defining the geotechnical characteristics of the surrounding country rock as well as the South Lobe kimberlite and involved the drilling, geotechnical logging and sampling of 35 diamond drill holes, totaling almost 22,000 m, with field and laboratory testing of the core samples. Acoustic Televiewer (ATV) logging was also conducted in a subset of holes to identify open joints and bedding planes and complement the oriented core logging data. A total of 10,886 tests were conducted on samples across the various lithologies, including:

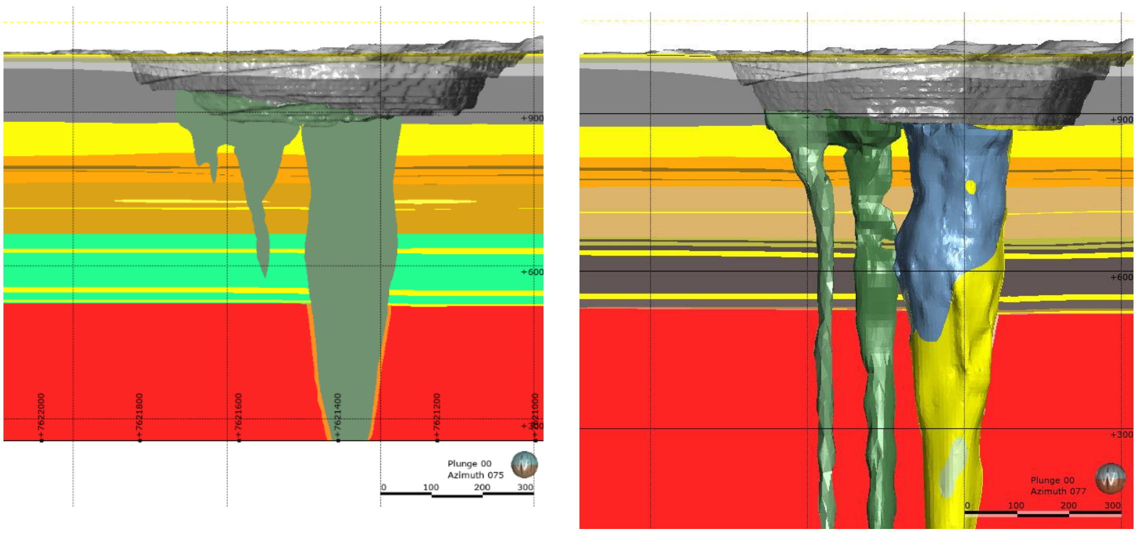
- Uniaxial compressive strength tests with Young's modulus & Poisson's ratio measurements (UCM);
- Brazilian tensile strength tests (UTB);
- Triaxial compressive strength tests (TCS);
- Direct shear tests on rock joints (SHJO);
- Rock base friction angle tests (BFA);
- Rock porosity tests (POR);
- Rock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

Key outcomes of the investigation program are as follows:

- Updating of the geological country rock, structural, and rock mass model based on the additional drilling (see Figure 16-3);
- Establishment of a detailed geotechnical logging database, including laboratory and field strength test results and structural orientation logs;
- Creation of a 3D rock mass block model that provides both statistical and spatial distributions of the project geotechnical data;

- Recording of core photographs from hyperspectral imaging program, which also provided the most reliable discernment of lithological contacts and detailed delineation of the weathering susceptible rock mass units; and
- Mitigation of several previously identified geotechnical risks.

Figure 16-3: The Country Rock Leapfrog model from January 2019 (L) and the Updated model (R), NNW-SSE section looking to ENE



Source: Pierce Engineering (2019)

16.3.3 Rock Mass Quality and Strength

The homogenous nature of the rock units at Karowe has resulted in geotechnical domains that closely follow lithology, with some additional subdomains (e.g. contact zones) established on the basis of lower intact strength. The unweathered granite basement host and south lobe kimberlite ore are both of very good quality, exhibiting high mean intact strength (UCS=137-146 MPa) and sparse jointing (>10 m spacing). This, combined with its low weathering susceptibility, makes the South Lobe kimberlite atypical. Kimberlite intact strengths are lower where the kimberlite is in contact with the country rock.

The bulk of the host rock above the granite, comprising approximately 345 m of sedimentary rock (shales, mudstones and sandstones of the Karoo Supergroup) and approximately 130 m of igneous rock (basalts of the Stormberg Lava Group) are of good quality, exhibiting intact strengths that are approximately half that of the granite and kimberlite (mean UCS=53-83 MPa) and similar sparse jointing (>10 m spacing).

There are some weaker layers within the country rock that exhibit low intact strengths (mean UCS=28-40 MPa). These include the upper Ntane sandstones, the red mudstone beds within the lower Mosolotsane sandstone, some layers within the Tlapana mudstones and the weathered granite. These last two units also have more tightly spaced joints (~1.2-4.4 m spacing, predominantly subhorizontal) than the remainder of the rock on site.

Rock mass classification indicates that the formations in the area of interest have fair to good rock mass quality. The average Laubscher RMR rating is between 50 and 60. The Q' of all lithologies except Kalahari ranges between 200 and 800, which is classified as extremely good to exceptionally good. The RQD for all the formations was 90% and above.

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

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

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

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

16.3.4 Weathering Susceptibility

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

The kimberlite did not demonstrate any susceptibility to weathering under wet-dry cycles due to its low clay content. The red mudstones of the Mosolotsane Formation were shown to degrade within one wet-dry cycle, while the mudstones, carbonaceous mudstones and coal layers of the Tlapana Formation exhibited a higher resistance, starting to degrade within three to five cycles. The Tlhabala unit is relatively competent and has a low susceptibility in general, with only a subset of samples exhibiting degradation. As a result, the rock mass strengths estimated for the susceptible subdomains in these units should be considered representative of in-situ strengths. Exposure of these materials to atmospheric conditions (in particular water) is expected to result in a greater than 50% reduction in their rock mass strengths within a short time. Any underground development that may take place in these materials should be sealed as soon as possible after exposure of the rock face to avoid degradation due to atmospheric exposure.

16.3.5 In-Situ Stresses

Analysis of regional tectonics suggests that in-situ horizontal stresses are low in the country rock (roughly half of the vertical stress). Estimates of the magnitude and orientation in-situ stress in the South Lobe kimberlite are based on wireline Sigra testing (overcoring method) completed by Sigra PTY Ltd. These suggest that the pipe has variable horizontal stresses, close to the vertical stress in the near-surface and higher than the vertical stress at depth.

16.3.6 Caveability

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

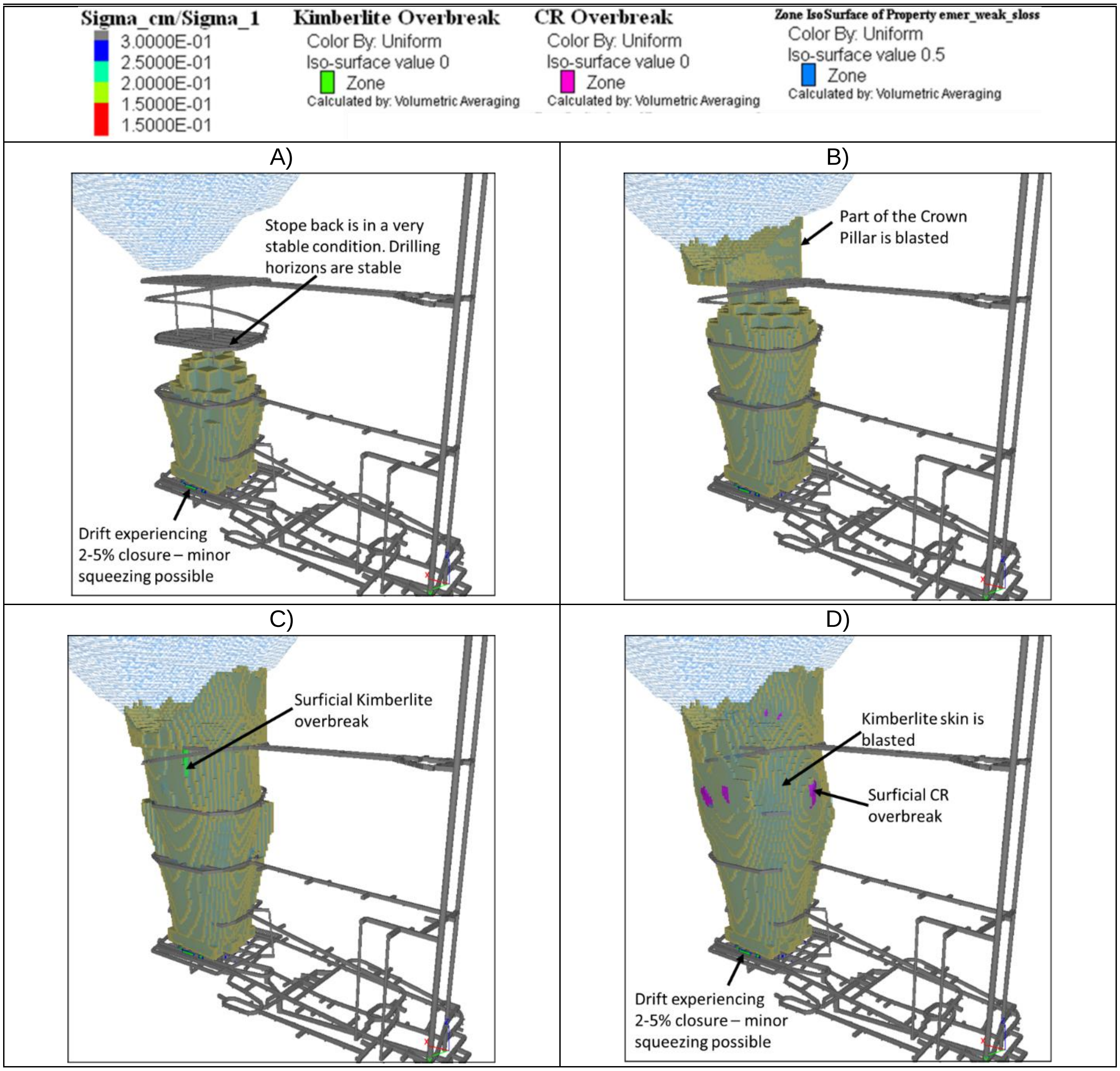
16.3.7 Brow and Crown Pillar Stability

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

The selected pyramidal sequence has the most stable back shape, which promotes stability with low overbreak and promotes stability of the crown pillar, which is predicted to have a factor of safety against collapse by the end of stoping of 1.3. In general, due to the high kimberlite quality and low in-situ stresses, stope overbreak of less than 5 m is predicted in general, with somewhat higher overbreak expected at weak internal zone / contacts.

Figure 16-4 illustrates the predicted overbreak and strength/stress ratio on development as stoping progresses with the pyramidal option with 15 m kimberlite skin. The semitransparent blue iso-surface shows where the rock has experienced damage and lost 50% of its cohesion.

Figure 16-4: FLAC3D forecast of Kimberlite and Country Rock Overbreak and Strength/Stress Ration on Development



Source: Itasca (2019)

16.3.8 Fragmentation

The fragmentation from stope blasting is expected to be manageable, with minimal oversize, based on the blasting results achieved in the pit at similar powder factors. Some larger blocks (>2 m^3) are expected to result from natural overbreak of stope brows but will be manageable with the large number of drawpoints and planned secondary blasting capabilities. Some minor to moderate attrition of oversize is also expected from secondary fragmentation during drawdown. The results of Rapid Emulator Based On Particle Flow

Code (REBOP) software simulations indicate that the percentage of fines expected at the drawpoint due to secondary fragmentation is ~10% and a reduction of oversize material in the order of 32% after drawing an equivalent 400 m height of draw.

16.3.9 Dilution Potential

FLAC3D analyses to date suggest that the potential for dilution of ore by overbreak into the surrounding country rock is very low due to the stabilizing effects of the pipe geometry (circular cross-section) but is sensitive to the assumptions around host rock in-situ stresses. The model results also suggest that the 15 m skin of kimberlite to be left against the host rock above the granite (to minimize potential for country rock overbreak entry / dilution and to improve stability) would be stable with a factor of safety against collapse greater than 3.0. The potential for dilution entry from pit wall failures after the crown pillar is blasted is considered low based on analyses to date but should be examined further once pore pressures are available for inclusion in the FLAC3D mechanical analyses of host rock stability.

16.3.10 Infrastructure Stability

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

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

16.3.11 Subsidence Potential

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

16.3.12 Hazards

The potential for mud rush is considered to be low given the high strength, low clay content and low weathering susceptibility of the kimberlite combined with the stabilization of clay-bearing sedimentary country rock offered by the kimberlite skin.

There is a low risk of seismicity due to the relatively low stress:strength ratios expected around development.

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

16.3.13 Recommendations

Additional predictive modelling is suggested to refine factor of safety estimates, including incorporation of evolving pore pressures from the hydrogeological model and varying rates of deterioration (of weathering susceptible layers) into the geomechanical model for the study of country rock overbreak, premature crown pillar collapse and pit slope instability following crown pillar blasting. In addition, the anisotropy in tensile strength should be refined within these models to better reflect domain-specific anisotropy ratios as current models conservatively assume zero tensile strength parallel to bedding.

16.4 Mine Water Control Dewatering Strategy & Design

16.4.1 Introduction

Exigo was appointed by JDS to conduct the hydrogeological site characterization, mine dewatering strategy and design for the KDM UG Project. One of the major risks identified in the PEA report, was mine dewatering, therefore this component of the FS was particularly important to detail out. The objectives were to characterize the hydrogeology and determine the mine water control, dewatering rates and mitigation required to manage the water risks.

16.4.2 Mine Planning and Scheduling

The open pit has been in operation since 2012 and is planned to end at an elevation of 710 masl (300 m depth). In 2021, the vent and production shaft work is planned to be initiated with underground mining beginning from 310 masl (310 L) in 2025. The mean open pit drop down rate is currently 24 m/year, which is anticipated to be accelerated towards the end of the pit life.

16.4.3 Hydrogeological Data Review, Gathering and Analysis

The sub-components that fed information to the LOM dewatering strategy and design consist of specialist reports, of which three are not yet available. The level of data gathered and analyzed is beyond FS requirements. KDM is a brownfields site with eight years (2012 to 2019) of actual mine dewatering data available on which the aquifer system behavior and pressure response can be analyzed and used in the model calibration.

16.4.4 Hydrogeology

The KDM is located in a semi-arid region. The geology consists of layered Stormberg Basalt, underlain by Ntane and Mosolotsane Sandstones that form a regional (main) aquifer. The main aquifer zone is underlain by Thlabala Mudstones and Thlapana Cabonaceous Shale Aquitards. The Tlapana overlies a weathered and solid / fractured granite.

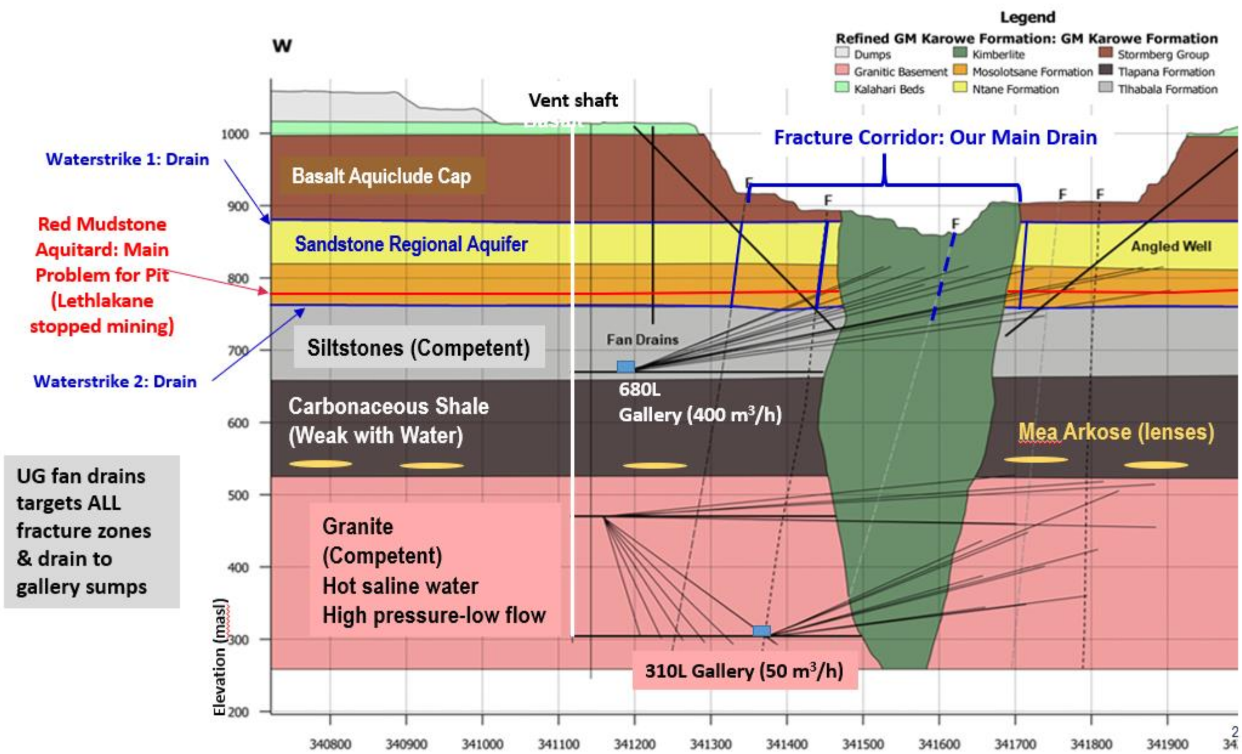
The open pit mine began development in 2011 and developed through the Stormberg Basalt into the upper parts of the Ntane Sandstone. The Stormberg Basalt-Ntane Sandstone contact forms a regional permeable aquifer zone. The main water bearing zones are shown in Figure 16-5 and are formed by:

- Basalt-Ntane contact, which forms the regional aquifer that is the source of water;
- The fracture corridor (NNW-SSE) which is linked to, and pressurizes by the Basalt-Ntane contact aquifer;

- The Mosolotsane base water strike (190 to 245 m depth) that is overlain and confined by the Red Mudstone Aquitard; and
- The Northern Kimberlite Pipe and contact, which is an inferred highly permeable zone that could form an important drain below the Mosolotsane-Thlapana contact.

Aquitards are formed by Grey and Red Mudstones at the base of the Mosolotsane Sandstone Aquifer and the Tlapana Black Shales. The aquitard zones are important as they have low permeability values and persistent head conditions. The Grey and Red Mudstones at analogue mines were responsible for hydrogeomechanical problems that led to pit wall collapse.

Figure 16-5: Karowe Hydrogeological Setting



Source: Exigo (2019)

16.4.5 Boreholes and Yields

Borehole yield is the flow rate that can be pumped from a borehole and is important as it relates directly to the mine dewatering potential and permeability of the subsurface. The mean borehole yield at which boreholes were tested before 2012 was 50.7 m^3/h. The tested yields ranged between 28 to 85 m^3/h. The vertical wells in the fracture corridor yields 15-25 m^3/h. The newly drilled angled and in-pit dewatering holes have yields of up to 60 m^3/h. Due to the confined to unconfined changes in the aquifer, borehole yields will drop by 30% to 50% and new boreholes will have to be developed to maintain the dewatering rates until the 680 L gallery and fan drains are installed.

16.4.6 Aquifer Parameters

Aquifer parameters from pumping tests that represent the Ntane & Mosolotsane Sandstone Aquifers had mean transmissivity values for the constant discharge tests ranging from 32 to 40 m^2/d. The packer test results showed hydraulic conductivity was variable and ranged from $2.27x10^{-5}$ to $5.47x10^{-1}$ m/d.

16.4.7 Piezometric Heads

The piezometric heads in the pre-mining phase were located at ± 935 masl (75 m depth), ± 25 m below the regional baseline groundwater levels, which were originally at ± 960 masl. The piezometric head declined by 75 m from 2011 to 2013 to 860 masl (150 m depth) where it stabilized at the Basalt-Ntane contact until August 2019. This stabilization effect occurred at an average pumping rate of ± 225 m^3/h. In September 2019, the dewatering rate was increased to 365 m^3/h, which influenced drawdown by a further +10 m in two weeks.

16.4.8 Hydrogeochemistry & Mine Residue Assessment

The natural baseline water quality from the regional Stormberg Basalt-Ntane contact water strike has a total dissolved solids (TDS) signature of 1,500 mg/L to 2,000 mg/L. The deep granites have saline water with 25,000 mg/L TDS. The upper Kalahari and weathered basalt zones do not form a continuous aquifer, as the regional groundwater level has lowered by 30 m since the 1970's. At Karowe, leakage from the storm water and TSF facilities causes localized perched conditions that seep to the open pit. The water quality that could be measured at shallow monitoring boreholes around the TSF was 5,000 mg/L TDS. This was the lowest value recorded. This means that based on TDS, the seepage would initially create a dilution plume, and later a localized elevated TDS signature of ±8,000 mg/L that would report to the open pit.

Arsenic is present in the TSF monitoring boreholes at 0.056 mg/L. This slightly exceeds the World Health Organization WHO (2017) limits for drinking water (0.01 mg/L). Arsenic is present at a concentration of 0.4 mg/kg in the whole rock (solid phase) and in the leach at 0.01 mg/kg (liquid phase), which is at the WHO (2017) drinking water limit. Arsenic will likely build up in the process water circuit over time. Two samples taken in 2018 and 2019 from the TSF return water confirmed arsenic concentrations below the detection limit (<0.006 mg/L). Modelling of arsenic transport from the TSF shows that the maximum calculated travelling distance after 100 years for the 0.01 mg/L limit, is 150 m.

16.4.9 Mine Dewatering Modelling Flow Rates & Piezometric Pressures

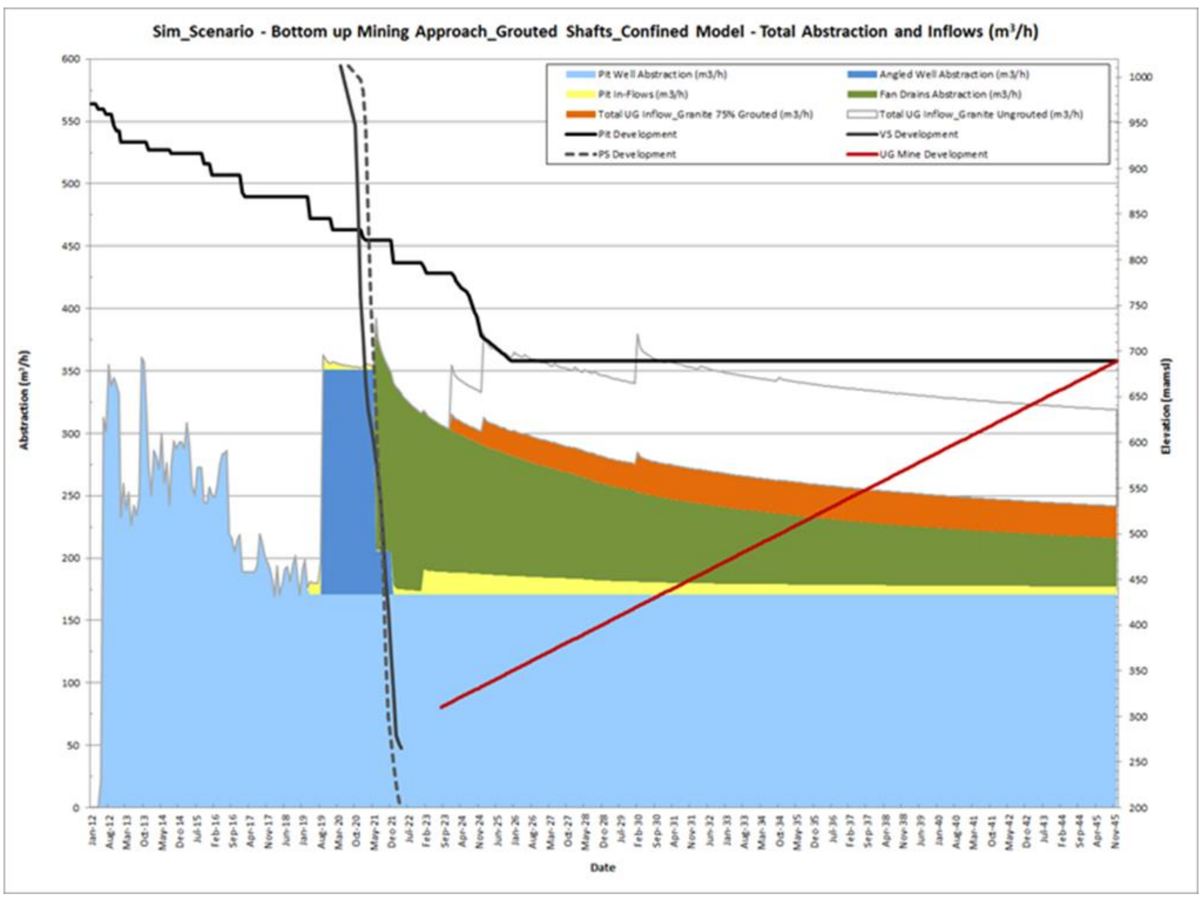
Mine dewatering modelling was done for the design scenario with shafts grouted. The simulations based on the 2011-2018 dewatering rates and head decline returned results with a transmissivity value range of 25-30 m^2/d and a storativity value of 0.001, which compares well with the mean aquifer parameters. For the design where the shafts were grouted, the modelling results show that:

- The yields of the current boreholes at 350 to 400 m^3/h will lower the piezometric head by ±60 m to 800-830 masl before the end of 2019. This will allow the open pit to develop until the underground gallery at 680 Level (680 L) is in place by January 2021.
- Due to the confined-unconfined behaviour, the dewatering rate of all the boreholes will decrease by 30% to 50% in the first 6-12 months between 175 m^3/h to 250 m^3/h. To mitigate this, provisions will be made to add additional pumping capacity to maintain the drawdown rate until the LOM UG gallery is operational.

- The 680 L underground gallery and fan drains have an important influence and are focused on creating two focused drain sinks at Target Areas A and B to the north and south of the Kimberlite Pipe, at the fracture corridor and Mosolotsane water strike junction. With the implementation of the 680 L fan drains, the inflow will shift at 375 m^3/h to the fan drains and will decline to 275 m^3/h in 2025.
- The piezometric heads and pore pressure distribution in the upper Sandstone Aquifer (Ntane & Moso) will decrease rapidly with the implementation of the 680L fan drains.
- The Red Mudstones in the open pit have persistent head and will take a long time to depressurize.

The confined model is transmissivity constant and therefore over-estimates flows. Unconfined calculations indicated that the dewatering wells and open pit inflows spike to 350 m^3/h and then decrease rapidly to 50 m^3/h when the underground fan drains are installed. The existing pit dewatering wells will lose most of their water to the 680 L fan drains. The dewatering rates will decrease to between 150 to 250 m^3/h in the Ntane & Moso Aquifers by 2035. Figure 16-6, Figure 16-7, Figure 16-8 and Figure 16-9 show the confined model results and schematics.

Figure 16-6: Confined Model: LOM Simulated Open Pit & Underground dewatering rates

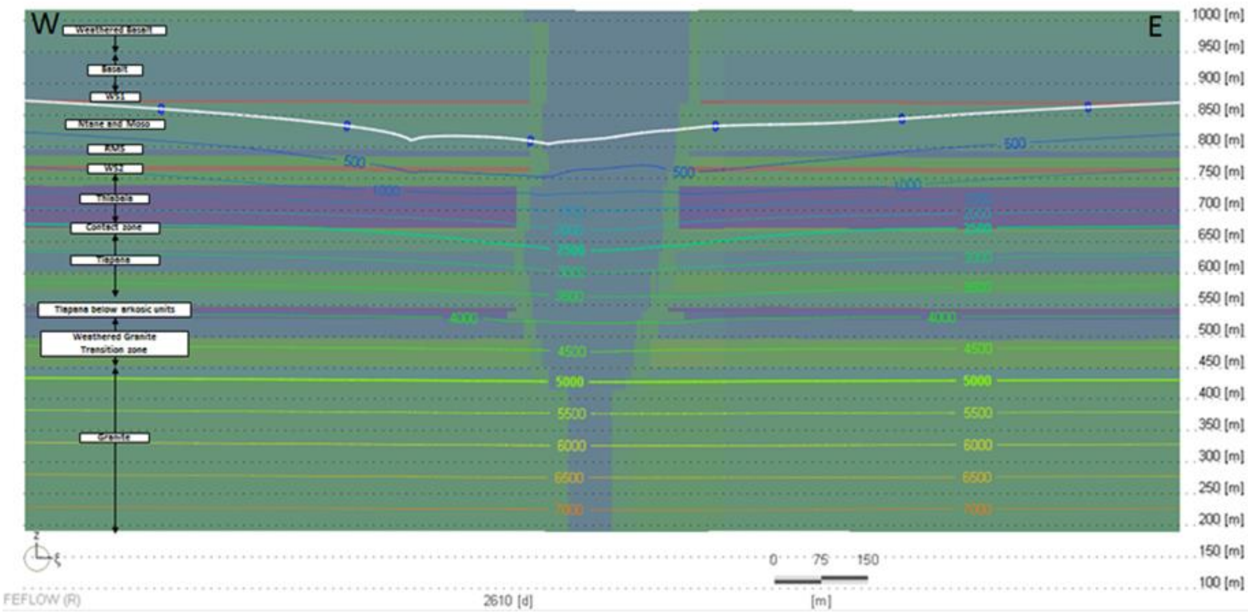


Note: 2018-2045 (shafts grouted design scenario)
Source: Exigo (2019)

16.4.10 Karoo Aquifer Behavior in Relation to Mine Dewatering

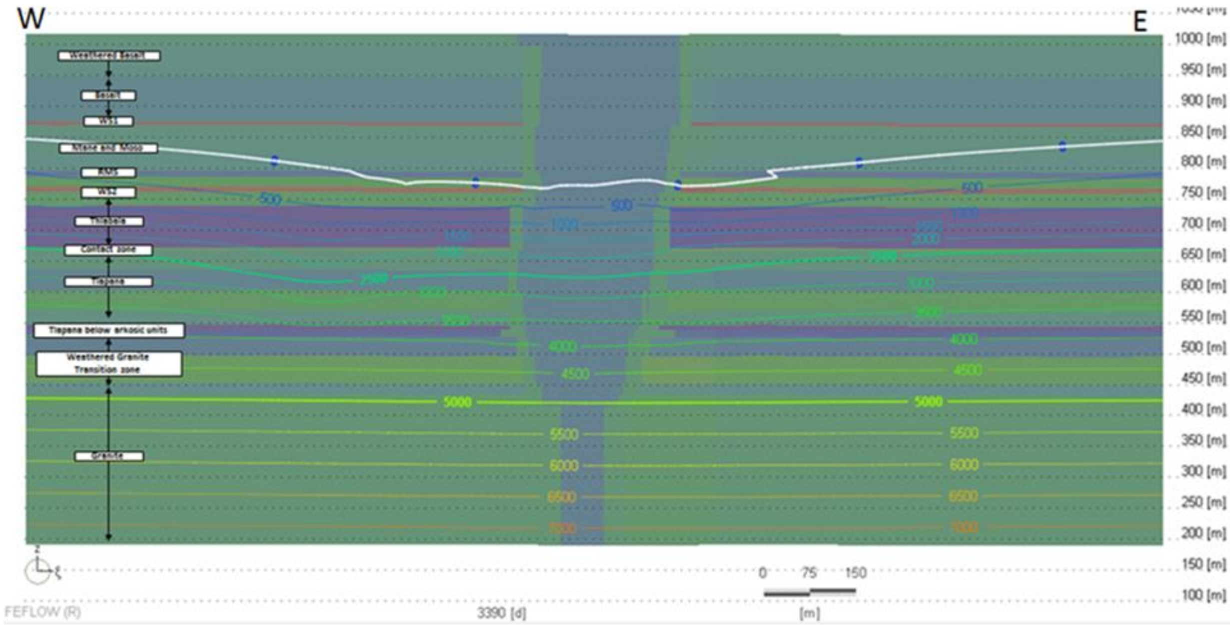
Due to head decline over time to a level below the aquifer top, the system will change from confined to semi-confined around the mine. With a continued drawdown and head drop, the transmissivity and flow rate will both decreases over time. The decrease in flow will be faster than the head drop and mine dewatering will become very difficult when using vertical wells. By using angled wells, the inefficiency can be reduced, but not eliminated. The LOM water control options to use an underground gallery and fan drains will mitigate this problem.

Figure 16-7: Confined Model: Simulated Pressure Distribution – April 2019 (calibration)



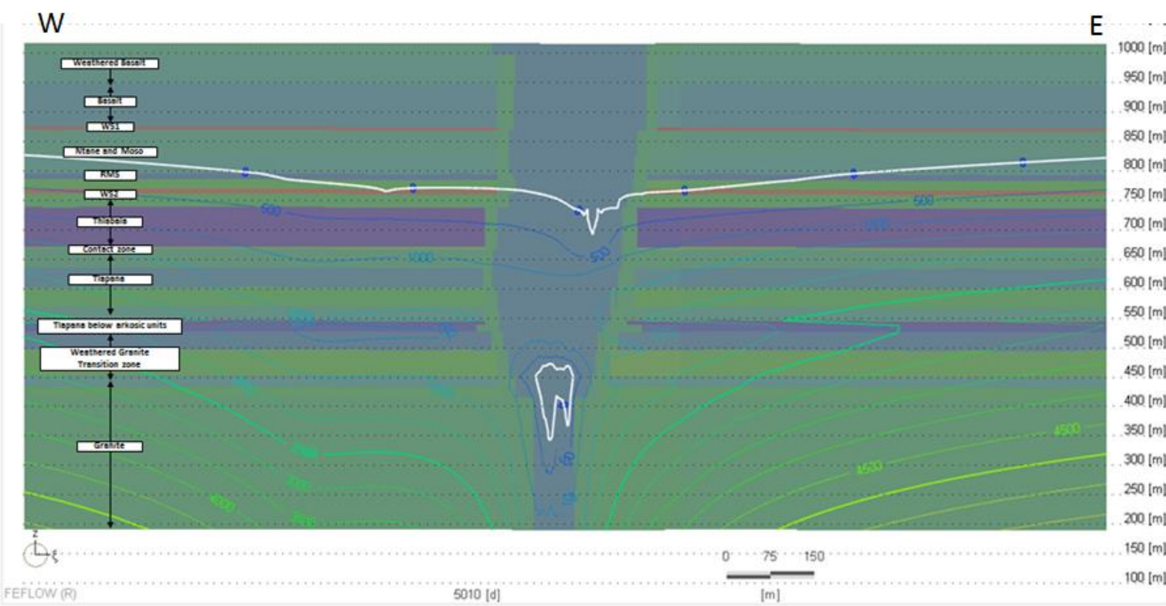
Source: Exigo (2019)

Figure 16-8: Confined Model: Simulated Pressure Distribution – Start of 680 L Gallery mid-2021



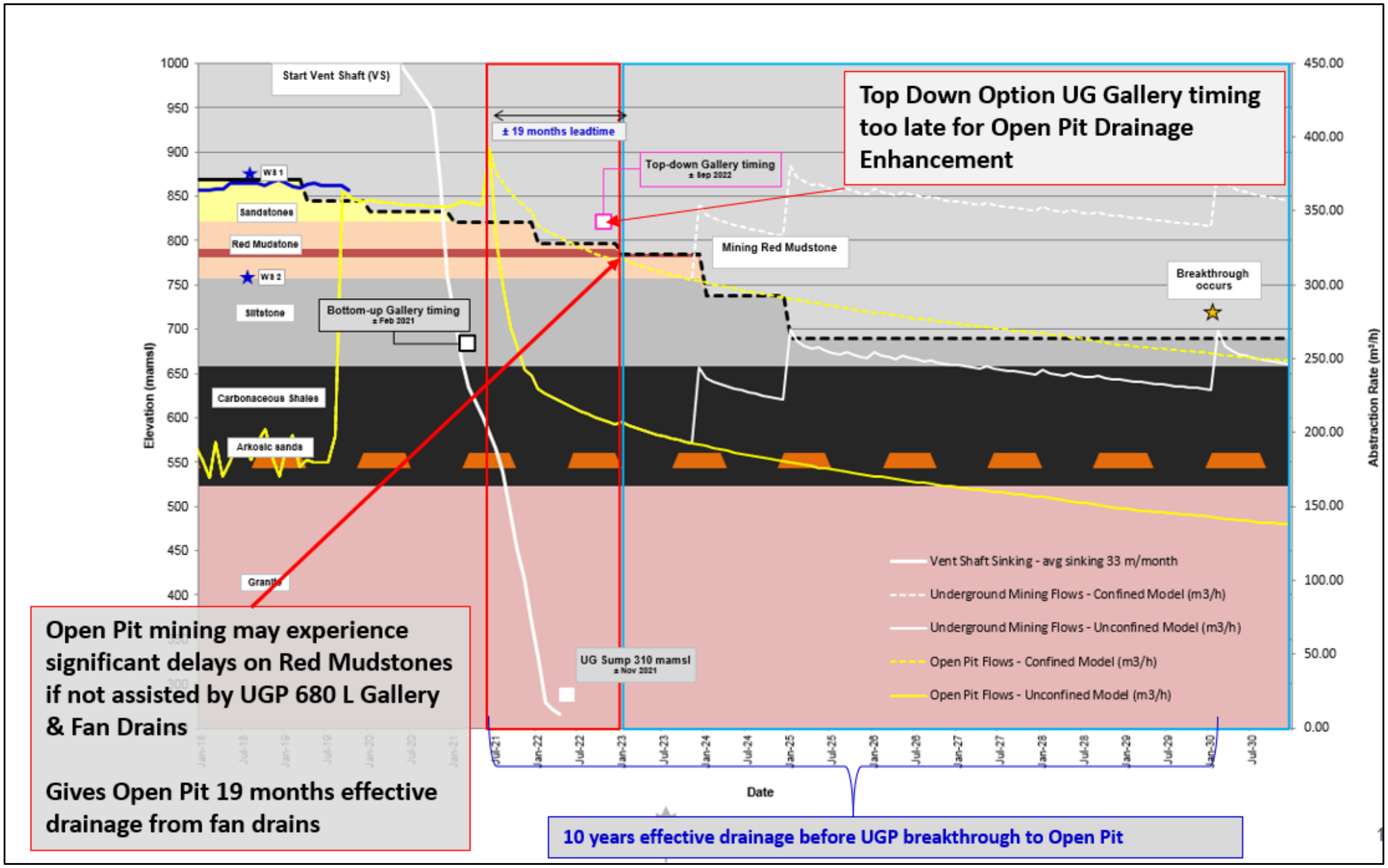
Source: Exigo (2019)

Figure 16-9: Confined Model: Simulated Pressure Distribution – End of OP 2025 & Start of UG



Source: Exigo (2019)

Figure 16-10: Karowe Open Pit and UG Mine Development Planning



Source: Exigo (2019)

The water control and dewatering strategy relies on a multiple design approach that relies on removal and isolation of water depending on the aquifer / aquitard system. The LOM water control strategy consists of (i) the grouting of shafts, the control of the deep saline granite water evaluation (grouting) holes and (iii) the 680 L & 310 L underground dewatering galleries with fan drains. The gallery and fan drains provide the best engineering solution which enables drilling at various angles to intersect the sub-fracture corridor and the fracture corridor at optimal angles to maximizing surface area and drainage towards a single point that can be managed. The technology is readily available in Botswana and Southern Africa and it allows for control and flexibility.

In terms of timing, the open pit requires the vent shaft, which enables the 680 L underground dewatering gallery and fan drains to be operational, 19 months ahead of the open pit intersection with the Red Mudstones Aquitard.

16.4.11 Benchmarking

Benchmarking is important to reduce the uncertainties associated with the dewatering parameters and strategy. There are two diamond mines in Botswana where mining was stopped due to ineffective water control. The models and aquifer behaviour were evaluated against the information from other diamond mines located in similar aquifer / aquitard conditions.

The dewatering strategy relies on design, timing and alignment with other critical components that are non-technical. Successful mine dewatering does not depend only on the hydrogeology; it functions as a system where hydrogeology, geotechnical, mine engineering and planning must be integrated towards and optimized. Implementation of the plan is key with a provision for ongoing review and optimization during implementation. The management and reinterpretation of hydrogeological dewatering data during implementation and recalibration of models are important.

Should the dewatering strategy and planning be followed with the quality control and assurance, dewatering can be implemented successfully within the defined time frames.

16.4.12 Recommendations

Recommendations related to dewatering include the following:

- This study should be updated and optimized once the geotechnical and structural geological models and data become available;
- The data gaps identified should be covered in the pre-construction phase;
- The dewatering pumping data must be measured on a weekly basis. Provision should be made for totalizers to support and reconciliate the scada data;
- The immediate dewatering acceleration program (IDAP) actions and installation of additional angled dewatering and an in-pit dewatering wells to keep the dewatering rates above 350 m^3/h must be monitored and reviewed on a monthly basis;
- The regional groundwater flow model and water supply model should be developed / updated and integrated with the localized mine dewatering model to inform the water supply assurance and cumulative impacts of other (all) diamond mines;

- Due to the fact that the groundwater head levels were almost stable in the past + four years, the mine dewatering model must be updated and recalibrated once three months of data at elevated abstraction rates and aquifer pressure response is available;
- The confined model must be updated with a confined to semi-confined to unconfined version;
- A more detailed risk assessment (source-pathway-receptor) should be done on the potential for build-up of arsenic in the process water circuit and migration towards the open pit should be reviewed with more detailed geochemical modelling to quantify the attenuation and precipitation limits; and
- The online water information management system (WIMS) should be integrated with the scada for real time dewatering status and pressure response reports.

16.5 Mine Planning Criteria

The mine planning criteria for the KDM project are listed below:

- The underground design and schedule will be developed as to maintain current mill feed rates and not consume surface stockpiles as the mine transitions from open pit to underground;
- The pre-production mine development period will commence in 2020 and take approximately five years to complete. The duration will be split between detailed engineering, surface preparations, underground access, level development, installation of the material handling infrastructure, and preparation of the extraction drawpoints;
- Underground commercial production will commence in Q1 of 2025 when production rates achieve a sustained rate greater than 80% of the 7,200 t/d target.
- All capital development, both lateral and vertical, is planned to be completed by local and expatriate mine contractors during pre-production, followed by owner operations during production;
- In-house labour currently operating the open pit will be utilized where able to support underground development and operations. Contracted training staff will remain on site through the mine life to transition open pit labour and support the underground operations;
- Conventional, trackless diesel and electric / hydraulic mining equipment will be utilized for construction and operations;
- Electric and automated Load Haul Dump units (LHD) will not be utilized in the mine, however, mine equipment will be equipped with the latest available technology to maximize performance and efficiencies; and
- Mined voids will remain open at the end of the mine life with no backfill required.

Other key mine planning criteria are summarized in Table 16-2.

Table 16-2: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	360
Shifts per Day	Shifts	2
Hours per Shift	Hours	12
Work Roster	On/Off	4/2
Nominal Ore Mining Average Rate	t/d	7,200
Annual Ore Mining Average Rate	Mt	2.6
Ore Density	t/m³	3.0
Waste Density	t/m³	2.9
Swell Factor	%	35

Source: JDS (2019)

16.6 Mining Methods

16.6.1 Open Pit

The open pit mine operation is expected to terminate mid-2025 at an elevation of approximately 710 masl. The mine currently has approximately two years of stockpiled reserves, which will be increased through the life of the open pit and then consumed according to value through the end of the mine life.

The average total (ore and waste) open pit mine production rate in 2020 is approximately 21,300 t/d. Production rates will decrease until the end of the mine life, as no further pushbacks are planned, and the strip ratio will be reduced at depth. Stockpile re-handle rates peak at the end of the combined open pit and underground mine life, when all mill feed will come from stockpile.

All open pit mining operations are performed by mine contractors working year-round on two 12-hour shifts. The on-site mining contractor is currently performing load and haul operations with a Caterpillar 6015 Hydraulic Shovel and Caterpillar 777E/G Haul Truck pairing. The mining contract has a mixed fleet of additional production, support, and ancillary equipment available on-site.

The Lucara Diamonds mining technical services team has provided the open pit mine design, production targets, and cost inputs used in the FS.

16.6.2 Underground

As previously discussed, the Karowe South Lobe is not expected to cave naturally. The lack of consistent horizontal stress will not produce consistent horizontal fracturing required for propagation of a natural cave. Due to the lack of horizontal stress, the use of preconditioning such as Hydro fracturing as used at Cadia East (Lowther et al., 2016) and other block caves around the world will also not work here. It is likely Hydro fracturing would lead to vertical fractures near the surface which would remain clamped together and thus not assist in cave propagation. Hydro fracturing could potentially cause horizontal fractures to develop at depth, however, these would not be consistent and thus unreliable from a cave propagation prospective.

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

Based on the factual data collected during the FS fieldwork as well as experience in the open pit, the mine plan favours LHS due to favourable risk profile, practicality and from economics.

Benefits of the LHS mining method include:

- Highest value ore to be extracted first due to the bottom up mining approach;
- Almost no development in weak, water-bearing lithologies;
- Dilution will be delayed (occurring after the payback period) as the weaker host rock is not exposed until later in the mine life. This is due to a combination of the mine method and the 15 m skin of kimberlite that will be left in the carbonaceous shale;
- Development and production of the underground mine can occur simultaneously with the open pit operations. With the production starting earlier than required, the reliance on the open pit stockpiles reduces and there is the ability to bring value forward with no impact on the production;
- Low operating costs;
- Ease of operation after the drilling and blasting phase is complete and small UG work force requirements;
- Early exclusion of surface water until the crown pillar is blasted;
- Greater control of ground water during development with grouted shafts;
- Significant ability to increase production after the drill and blast phase is complete;
- The mine design is set up with an extraction level; and
- Designed to manage natural caving should it occur.

16.7 Mine Design

The KDM underground mine design is based on a panel or block cave layout. Similar to block caves, the KDM design includes a main extraction level at the bottom of the mine workings from which all production ore is mucked.

The extraction level is designed with an offset herringbone layout to provide maximum mucking flexibility and protection from mud rush. In the event that the Lobe should start caving naturally, coarse fragmentation is expected due to the lack of jointing and structure within the lobe. The size and spacing of drawpoints on the extraction level is designed to manage this event.

The proposed design allows for maximum draw control of the blasted ore, whereby operations will utilize numerous draw points to manage the shape of the muck pile and reduce preferential draw of dilution. The design allows for continuous mucking to keep the muck pile in motion at all times, minimizing risk of re-compaction or creating a deadweight above the extraction panels. Constantly drawing from each drawpoint minimizes the risk of a mud rush or water rush by mixing any pockets of water that may have developed within the muck pile with dryer material.

Storage capacity has been designed into the mine plan to allow for constant movement of material from the drawpoints in the event of a material handling shutdown (planned or unplanned). The storage capacity has been designed such that 6 buckets of material can be draw from each open draw point per 24-hour period for up to 10 days.

Longhole drill horizons have been designed for the drilling and blasting operations required for this mining method. The shallow country rock lithologies (sandstones, red mudstones and carbonaceous shale) are of concern for increased dilution. To mitigate the risk of early dilution and preferential drawing of the country rock over the blasted kimberlite, a 15 m kimberlite skin embedded into the granite country rock has been left in place until the end of the mine life. This kimberlite skin will be extracted with the final blasting of the crown pillar, thus delaying and possibly eliminating the risk of high levels of dilution.

Mine design and scheduling was completed in Deswik software. Stopes were designed using Maptek© Vulcan 3D software. Geovia PCBC software was used to select the optimal extraction level elevation. Itasca developed a drawdown simulation using REBOP and Flac 3D software which was used as guidance to schedule underground production from the mine. An Arena simulation was completed by SRK to validate production rates selected for the operation based on drawpoint layouts, mobile equipment type and size, crushing capacity, ore storage capacity and hoist capacity.

16.7.1 Mine Access

There is currently no existing underground access at KDM. Aside from the open pit, the topography at KDM is relatively flat with no ability to take advantage of natural gradients for adit development.

Access to the underground mine will be from a 765 m deep production/service shaft, 7.5 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 a small auxiliary cage for personnel. This shaft will also serve as the main fresh air intake to the mine. A second shaft, 6.0 m in diameter, 715 m deep, sunk from surface to 295 masl, will be equipped with a heavy lift hoist for moving large equipment throughout the mine and hoisting development waste during pre-production development. This shaft will serve as the main exhaust route and secondary egress for the mine.

Shafts will be sunk blind using conventional drill and blast equipment and developed concurrently. Average sinking rates range from 1.2 m/d during the P/S pre-sink up to 2.5 m/d in the smaller vent shaft through good ground. It is expected to take approximately three years to fully sink and equip both shafts, plus another two years to complete all underground development, capital installations, and production ramp up. There will be a total of eight working levels in the mine, three of which will be accessed by a shaft station and the other five from internal ramps.

16.7.1.1 Ramp vs. Shaft

Access to the underground mine was decided based on several design factors including:

- Depth of resource;
- Mining direction;
- Production rate;
- Geotechnical criteria;

- Hydrological criteria; and
- Mine Life.

A decision matrix was generated to help decide on mine access method. Table 16-3 outlines the results of the matrix.

Table 16-3: Mine Access Decision Matrix

Consideration	Comments	Score	
		Ramp	Shaft
Resource Depth	+700 m		1
Continuous to Surface	Yes	1	
Mining Direction	Bottom-Up		1
Production Rate	7,200 t/d		1
Geotechnical Criteria	Poor conditions in stratigraphic units		1
Hydrological Criteria	Large inflows in stratigraphic units		1
Mine Life	+10 Years		1
Total Score		**1**	**6**

Source: JDS (2019)

The depth of resource is typically the first element of consideration with mine access. Resources with a depth of greater than 700 metres are proponents for shaft access, especially if the resource is not continuous to surface. The South Lobe extends over 700 metres below surface.

The selected mine method utilizes a bottom up approach. Although ramp access is possible, it would need to be driven to the bottom of the stoping horizons prior to start of commercial production which is time and cost prohibitive.

The production rate, at over 7,000 t/d would require the installation of at least two ramps which becomes economically unfavorable when compared to shaft access.

The geotechnical challenges associated developing through the red mudstones and carbonaceous shale, further discussed in Chapter 16.3 significantly impact conventional drift and ramp development rates and cost. Ramping through this material would negatively impact capital cost and schedule, while shaft sinking is better equipped to maintain high rates and reasonable costs through poor ground conditions.

A variety of high flow aquifers with large anticipated inflows may require grouting of both shafts and ramps until the water table is drawn down or development has advanced through the wet ground. The time and cost of this exercise, and impact to development rates is amplified in ramp access which achieves 15% of the vertical advance per metre developed compared to a shaft. The shaft will allow for relatively quick access to dry ground below the water table, from which drain holes can be installed to dewater the surrounding area.

16.7.1.2 Shaft Design Criteria

The UMS Group was retained by JDS to design and schedule the required shafts. UMS Group provides technical, advisory and contracting services through their Mining Engineering Technical Services (METS) Mining and Shaft Sinkers companies.

Two shafts are required for KDM to meet UG production demands and ensure the mine remains fully ventilated. The shafts are located approximately 375 m northwest of the South Lobe and 100 m from each other; their location was determined based on:

- Available geotechnical information and supporting drilling data;
  - Geotechnical holes have been drilled to test, understand, and predict the geotechnical properties of the lithologies to be encountered by the proposed shaft locations. See chapter 16.3 for details.
- Avoidance of the potential subsidence zone;
  - The geotechnical work carried out, as discussed in Chapter 16.3, indicates that the inherent stability of the Lobe shape will not cause any significant subsidence. The final excavation shape or subsidence zone of the cave is expected to remain within metres of the actual Lobe shape.
  - 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 open pit operation; and
  - The shaft locations were placed a minimum of 150 m outside of the final pit walls of the open pit design.
- Available landscape.
  - The site is already well established with infrastructure including waste dumps, ore stockpiles, processing facility, fine and coarse residue deposition facilities, dewatering wells, camp, and roads. Existing infrastructure was avoided as part of the shaft design criteria.

Production - Service Shaft: Design Capacity

The Production-Service (P/S) shaft has a rock hoisting capacity of 3.2 to 3.5 Mt/a. This is based on availability and utilization (this capacity excludes any downstream material handling constraints) of the two 21 t skips.

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

- Two skips with payload capacity of 21 t each;
- The cross-sectional area of 2.4 x 3.1 m;
- The service hoist consists of the cage and counterweight;
- P/S shaft auxiliary cage with an auxiliary hoist which can be used as a second means of egress in lieu of a manway; and
- The skips cages and counterweight are all operating on fixed guides.

The P/S shaft will serve as primary egress during the pre-production and production period. During the pre-production period development rock will be hoisted to surface through both shafts until the P/S shaft is equipped for permanent operation. When this takes place the development rock will be hoisted in the ventilation shaft. All services such as fresh water, compressed air, concrete supply and dewatering lines will be installed in the shafts.

Ventilation – Heavy Lift Shaft

The ventilation shaft consists of an auxiliary cage and single drum hoist which will be used for a second means of egress. In addition, the shaft will be equipped with a crosshead running on rope guides for purpose of hoisting a maximum payload of 60 t. This will accommodate the hoisting of heavy mobile and fixed equipment.

The P/S shaft will act as the fresh air intake throughout the pre-production and production periods. The ventilation shaft will remove exhaust air from the underground mine. Shaft diameters were selected based on hoisting capacity and ventilation requirements. Table 16-4 summarizes the ventilation criteria for the shafts.

Table 16-4: Shaft Ventilation Criteria

Criteria	Units	P/S Shaft	Ventilation Shaft
Maximum Velocity	m/s	9.1	16.3
Maximum Airflow	m^3/s	340	350
Shaft airway resistance factor	x 10-4 Ns2/m4	125	75

Source: JDS (2019)

16.7.1.3 Shaft Sinking Methodology

The two shafts at KDM will be blind sunk using conventional drill and blast techniques. The production and ventilation shafts both require a pre-sink. The pre-sink phase allows for a suitable shaft depth of 100 m to be established to accommodate the main sink shaft equipment and allows the sinking process to commence as soon as possible while the main sink infrastructure is being fabricated and installed.

The initial 50-75 m deep pre-sink phase at KDM will be completed using a heavy lift mobile crane. Materials and equipment will be lowered to the shaft bottom and mined material will be extracted via a kibble. A hydraulic excavator with multi tool function will be used to drill the shaft explosive rounds and also ground support holes in the sidewall. The following sequence is utilized during the pre-sink phase:

- With the drilling attachment, the excavator blast holes are drilled, charged and blasted;
- Kibbles are then loaded with the blasted material. The kibbles are then lifted to surface and tipped;
- The excavator will then complete support drilling. Mesh, secured on by split sets and washers, will be installed on the sidewall; and
- A blow over of the shaft bottom is completed before the next drilling phase to ensure all remaining explosive material is removed.

Once the shaft reaches a depth of 35 m, a drilling jumbo rig will be used to drill longer and faster explosive rounds. The excavator will continue being used for the shaft sidewall support.

Once the pre-sink phase is completed, the shaft will change over to the main sink phase. This will take the shaft down to its final depth and is capable of faster sinking rates, installing services and establishing infrastructure in the shaft. The following tasks are to be completed during the main sink phase:

- Kibble winder construction;
- Stage winder construction;
- Emergency winder construction;
- Shutter winches construction;
- Mucker winch construction;
- Main sink stage installation;
- Bank steel installation; and
- Sinking headgear construction.

The headgear used for the sinking, will remain as the permanent headgear for shaft operations. A grout curtain will be utilized to minimize water inflow.

Joy VSM-14 (Cryderman) shaft muckers will be utilized for the lashing during the main sink.

Once the shaft has reached its final depth, installation of the shaft steelwork is done from the top down. Steelwork will be completed only after sufficient preproduction level development has been completed so that the shutdown of the P/S shaft for equipping does not impact the schedule to complete the preproduction development.

16.7.1.4 Shaft Station Development & Infrastructure

All shaft station development will be developed from the ventilation shaft to avoid having to sling equipment down both shafts. The P/S shaft will break through to the shaft station to establish a connection. The shaft development crew will be responsible for establishing shaft stations and sufficient development to support the lateral development contractor.

16.7.1.5 Shaft Equipping

Equipping of the shafts takes approximately 8 months to complete. The following events will take place:

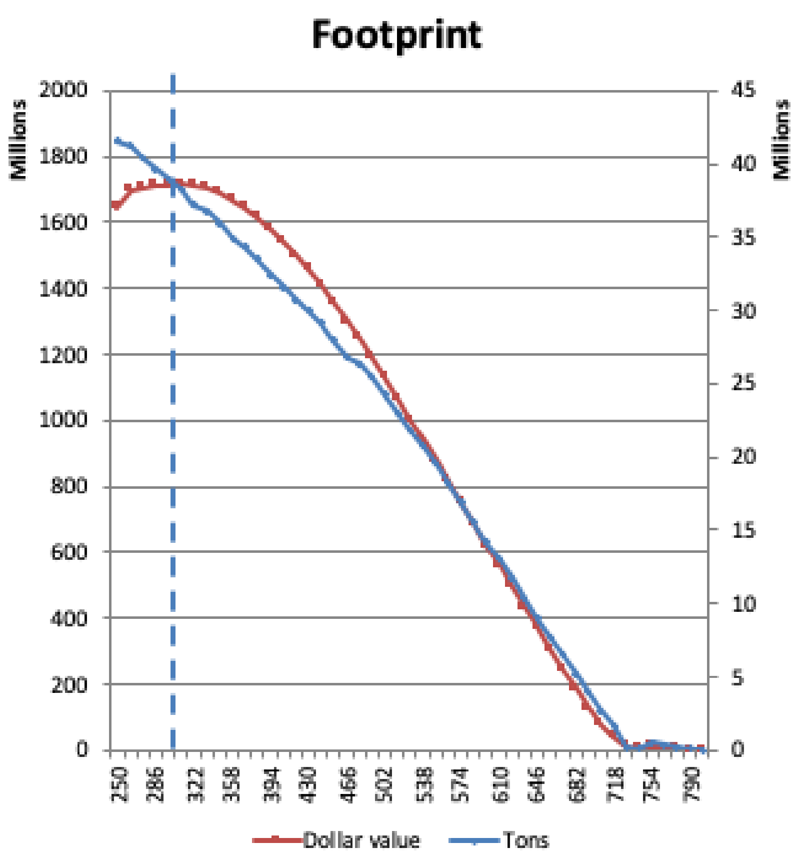
- Equipping loading pocket and loading station;
- Remove sinking services and install permanent pipes and power/communications cables from the shaft collar to the mine levels;
- Change headgear, sub bank and bank to permanent set up;
- Install equipping workstages;
- Equip shaft barrel from sub bank to shaft stations. Install station steel, brattice wall and screens to shaft stations;
- Strip and remove workstage;
- Remove stage ropes;

- Install permanent wire ropes;
- Install permanent conveyances and skips; and
- Commissioning of the system.

16.7.2 Extraction Level Optimization

Geovia PCBC suite of caving software was used to determine the most economic extraction level based on the 2018 resource model. PCBC footprint finder is an optimization tool that determines the most economic extraction level of the deposit by taking into consideration tonnage, elevation, and rock value. The most economic footprint elevation is the intersection of the greatest tonnes with highest rock value, as shown in Figure 16-11.

Figure 16-11: Footprint Finder Optimal Extraction Level



Source: JDS (2019)

16.7.3 Production Rate

The following factors were considered in the estimation of the underground mine production rate:

- Existing mill capacities;

- Drawbell productivities; and
- Sequence of mining and stope availability for drilling and blasting.

An underground mine production rate of 7,200 t/d was selected for this FS study and validated through an Arena simulation. The production rate is considered appropriate due to size of the orebody and the selected mass mining method.

16.7.3.1 Arena Simulation

SRK Consulting (Canada) Inc. (SRK) was retained by JDS to complete an Arena simulation for KDM. The primary purpose of the simulation model was to test the ability of the LHD fleet and material handling system to achieve the production targets. The model was also used to find bottlenecks, quantify equipment requirements, and identify the maximum capacity of the system.

The simulation incorporates mobile equipment such as production level LHDs and secondary breaking equipment, and infrastructure including crushers, bins, underground conveyors, and skips.

The simulation was used to determine the minimum LHD fleet size required to reach a target of 7,400 tonnes per day. Fleet sizes were increased until the following conditions were satisfied:

- Achieving a minimum average of 7,400 tonnes per day for a full year;
- Achieving a minimum daily crosscut target tonnes per day for a full year in all crosscuts; and
- Maintaining manual LHD engine hours below 4,600 per year per unit.

The design was able to meet the extraction criteria using three LHDs dedicated to the production level. Peak LHD production of 9,250 t/d was achieved using manual LHD operations, and 10,360 t/d with autonomous LHD operations. The skip hoisting system peaks slightly above 10,360 t/d.

16.7.4 Underground Development Criteria

A minimum 1.0 m distance on either side of mobile machinery has been used to size development headings. Additional considerations for ventilation may dictate larger heading requirements to meet maximum allowable air velocities.

A 21 t LHD machine is the largest piece of mobile equipment planned for use underground and requires a minimum 5.5 mW x 5.5 mH heading as recommended by the manufacturer. All areas of the mine which are to be regularly accessed by the 21 t LHD have been sized to a minimum 5.5 mW x 5.5 mH.

Where a 21 t LHD is not to operate regularly, minimum heading dimensions have been sized to 5.0 mW x 5.0 mH to accommodate the largest piece of development machinery, a 17 t LHD.

A minimum 3% gradient has been applied to all lateral development to ensure mine water reports away from the working areas and towards the appropriate sump. Maximum ramp gradients of 15% have been applied where mobile equipment is required to tram regularly under load. An exception to this is the conveyor drive which will be driven at 17% gradient.

All development is considered long term (in use for more than one year) and will incorporate a 1.0 m radius arched back, except for safety bays which will have a flat back. Development profiles and gradients are shown in Table 16-5.

Table 16-5: Underground Development Criteria

Development Heading Parameters	Width (m)	Height (m)	Maximum Gradient (%)
P/S Shaft	7.5 Ø	n/a	90
Ventilation Shaft	6.0 Ø	n/a	90
Loading Pockets	7 Ø	n/a	90
Raisebore Raises	3 Ø	n/a	90
Drop Raises	4.0	4.0	90
Maintenance Shop, Loading Pocket Drift	6.5	8.0	3
Conveyor	6.0	6.0	17
Ventilation Drives	6.0	6.0	3
Ramps	5.5	5.5	15
Shaft Stations, Access Drifts, Drawpoints, Panel Drifts, Explosives Magazine, Extraction Drive, Drilling Station and Remucks	5.5	5.5	3
Upper Drilling Horizons, Pump Station, Storage, Refuge bays, Substation, Fueling bays, Concrete Transport and Sumps.	5.0	5.0	3
Safety Bays	2.0	2.0	0

Source: JDS (2019)

Ventilation drives provide access to the extraction area, maintenance shop, explosives magazine, conveyer, crusher and loading pockets. Ventilation drives will be developed 6.0 mW x 6.0 mH to meet maximum allowable air velocities.

Remuck bays (remucks) will be excavated on the ramps and ventilation drives to reduce the development mucking cycle time. Intersections may be used temporarily as remucks during development, and where not available dedicated remucks will be developed at 150 m intervals. Additional remucks will be developed in the extraction area to provide storage capacity during periods of crusher maintenance. Remucks, in conjunction with other nearby infrastructure will be developed to store approximately 10 days of production. This will ensure constant movement of the muck pile and reduce potential for water collection within the draw bells.

Water collection sumps will be located at every shaft station and near production zones. Sumps will be strategically placed at low areas of the mine and prior to decline development to minimize reliance on ditching and redundant pumping. Major sumps and pumping equipment will be developed to manage a 10,000 m^3 per day inflow during storm events.

Sump water will be directed to one of two main pumping stations located approximately 350 vertical meters apart. The upper pump station will provide capacity for a major dewatering program prior to production and later be used as a booster station to pick up water from the lower pump station and direct to surface. Pump stations are located near the shafts.

Substations will be located at the shaft station on each level to provide power for the mobile equipment and the primary ventilation and dewatering infrastructure. Additional power centers will be located adjacent to major power draws such as production zones, maintenance facilities, crusher room, conveyor, and cooling equipment.

A permanent refuge station cut-out will be located on the main extraction level near the fresh intake air. Portable refuge chambers will be situated on all active working levels and provide sufficient capacity for all persons working in the vicinity.

Safety bay cuts outs sized at 2.0 mW x 2.0 mH x 2.0 mL will be driven every 15 m on corners, 23 m on straight ramp, and 30 m between any straight sections. Safety bay spacing adheres to Botswana Mine Regulations.

Internal intake and exhaust raises will be used to bring fresh air into the extraction area and exhaust air towards the ventilation shaft. This will ensure a constant supply of fresh air to the main working area. Raises greater than 30 m will be driven by a raise bore machine, and those less will be done with a long hole drill.

A raisebore machine will drive 3.0 m diameter raises within the kimberlite to serve as production slot raises, development muck passes, and fresh air ventilation between working levels. Raises will be driven in multiple sections from the main extraction level to the topmost drill horizon, and to surface within the open pit.

Drawpoints will be developed in a manner which will lend themselves to both shrinkage stoping as well as block caving. Table 16-6 outlines the drawpoint design criteria.

Table 16-6: Drawpoint Design Criteria

Development Heading Parameters	Unit	Value
Layout Type		Herringbone
Drawpoint spacing	m	18.0
Panel spacing	m	30.0

Source: JDS (2019)

16.7.5 Underground Mine Development

The production and ventilation shaft will have multiple shaft stations to provide access to various levels throughout the mine. Table 16-7 summarizes the shaft stations.

Table 16-7: Shaft Station Elevations

Level	Name	P/S Shaft	Ventilation Shaft
680 – Drilling Horizon	680 L	Personnel access to the drilling horizon.	Equipment access to the drilling horizon. Return air enters here.
480 – Drilling Horizon	480 L	Personnel access to the drilling horizon.	Equipment access to the drilling horizon. Return air enters here.
335 – Top of Loading Pocket	335 L	n/a	Return air enters here. Personnel access to this level is from the 310 level.
310 – Extraction Level	310 L	Personnel access to the main extraction level.	Equipment access to the main extraction level. Return air enters here.
285 – Bottom of Loading Pocket	285 L	Skip loading from the loading pocket bins. Personnel access to this level is from the 310 level.	n/a

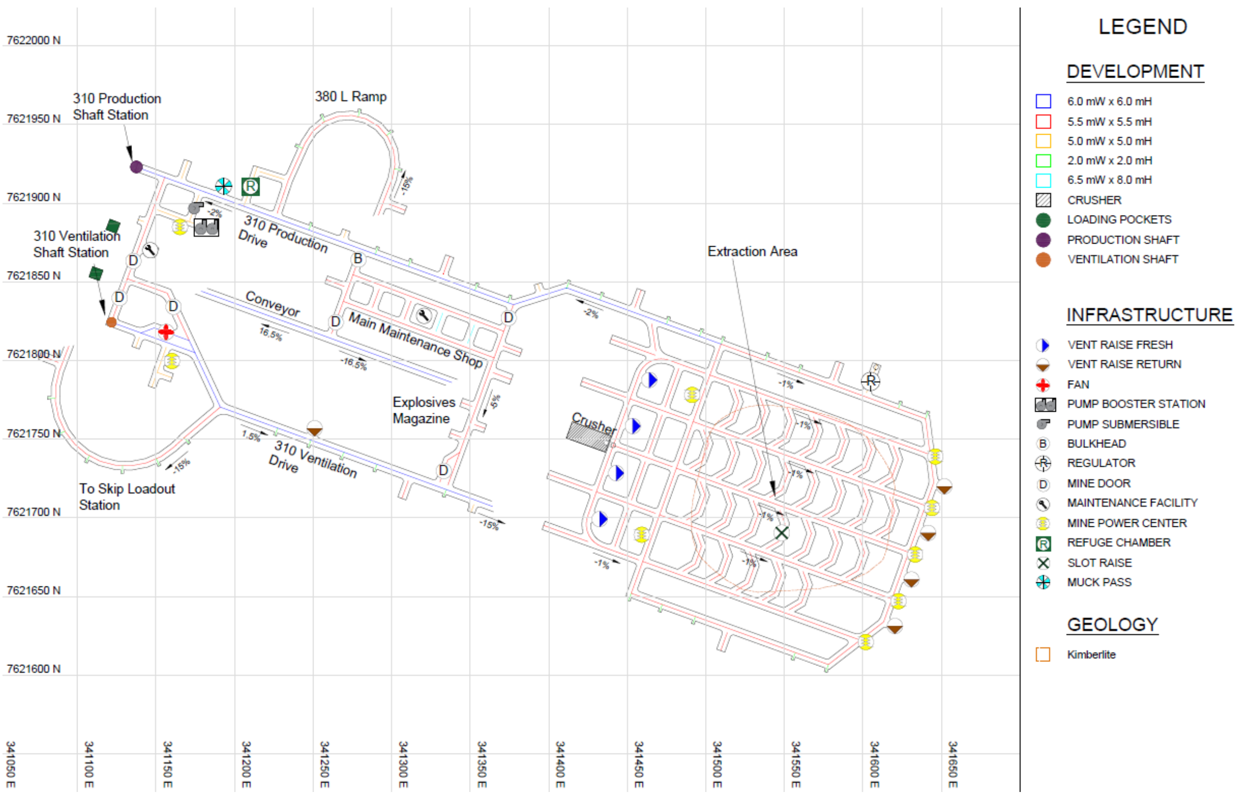
Level	Name	P/S Shaft	Ventilation Shaft
245 – Ventilation Level	245 L	Lowest part of the mine, a sump is located here and pumps water to the 310 pump station. Personnel access to this level is from 310 level.	n/a

Source: JDS (2019)

16.7.5.1 310 Extraction Level

The extraction level is located at 310 masl (L) and is accessed from both the P/S shaft and the ventilation shaft. Personnel access will be through the P/S shaft and all heavy lifting will be performed by the ventilation shaft. Figure 16-12 shows a plan view of the 310 L.

Figure 16-12: 310 L Plan View



Source: JDS (2019)

From the P/S shaft there is direct access via the 310 L production drive to the 380 L ramp, maintenance facility, explosive magazines, conveyor drive, and the production drawpoints. Near the P/S shaft will reside the lower pumping station, sub-station, concrete slick line, storage facilities, and main refuge and lunchroom. The 310 L will be driven at a +2% gradient to direct water inflow towards the shaft station sumps.

From the ventilation shaft, there is direct access, via the 310 L ventilation drive, to the skip loadout station, crusher room, main sump, and to the bottom of the P/S shaft. Near the ventilation shaft will reside the main exhaust fans, power supply, and chillers for mine air cooling.

A crosscut between the two shafts will provide initial ventilation to the level during development, and a bypass drive equipped with air doors will allow for man and equipment access between the shafts during production without disrupting ventilation. A substation will be installed between the two shafts that will provide power for the main fans and pump station.

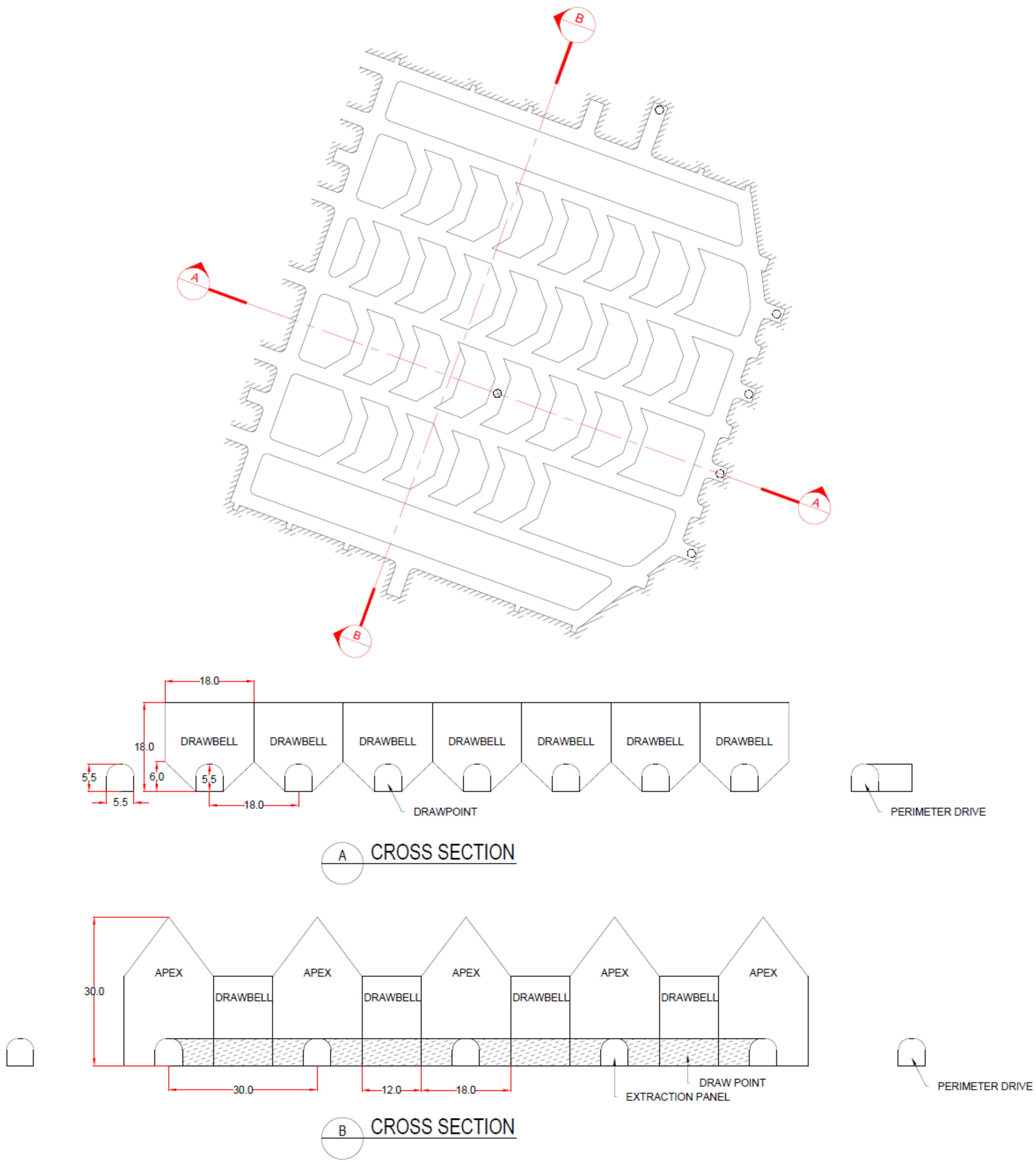
A second crosscut will join the 310 L production drive with the 310 L ventilation drive approximately 210 m from the shaft station to establish a second ventilation connection to aid in level development. This crosscut will also be equipped with air doors to allow for man and equipment access during production without disrupting ventilation on the level.

The 310 L production drive will terminate at the North side of the extraction area where LHDs muck from drawpoints and deliver ore to the crusher grizzly. The extraction area consists of five panel drifts spaced 30 m apart, equipped on both sides with a series of drawpoints spaced 18 m apart. A total of 54 drawpoints will be developed throughout the five panels as illustrated in Figure 16-13. A perimeter drive will be developed around the panels to provide a bypass as well as access for service equipment and supervision. The extraction panels will culminate on the west side of the extraction area to access a fixed grizzly and rock breaker which feed an underground jaw crusher coarse ore bin located below the level.

Fresh air is supplied to the level through four Fresh Air Raises (FAR), 4 mW x 4 mH in size, and the 310 L production drive that connects to the P/S shaft. Fresh air enters the level on the West side and travels through the panels where it is exhausted through four exhaust raises, 4 mW x 4 mH in size, on the East side.

Additional remuck bays and storage will be driven to accommodate muck storage during periods of maintenance on the material handling system. There are 250 m of remucks surrounding the crusher available for short term storage.

Figure 16-13: Drawbell Geometry



Source: JDS (2019)

16.7.5.2 380 L Drill Horizon

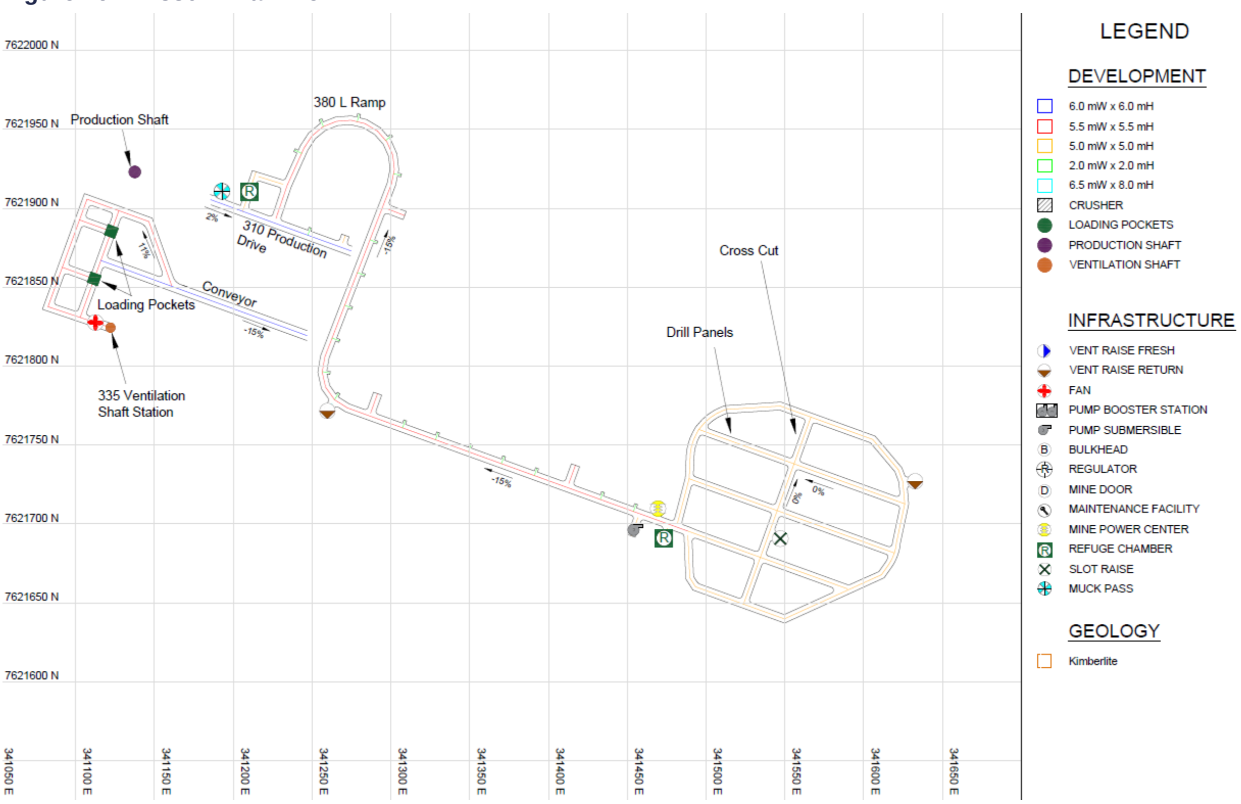
The 380 L drill horizon will be accessed via ramp from the 310 L and serve as the first of four drill horizons used to drill and blast the stopes.

Drill panels will be excavated on 30 m spacing across the South Lobe. A central crosscut will be driven perpendicular to these panels to serve as access for the slot raise required to start stoping. An additional drift will be driven around the circumference of the South Lobe to connect each drive together and provide access to the far end of the drill horizon once the central slot has been excavated. Four parallel drill panels are planned on the 380 L drill horizon.

A temporary raise will be driven halfway along the access ramp to provide an exhaust route and establish a ventilation circuit during level development. A second raise will be installed on the east side of the level to provide a permanent exhaust route and flow through ventilation on the level. This raise will report to the exhaust drive. Once the second exhaust raise has been installed the first may be barricaded or used as a muck pass to support panel drifting.

Figure 16-14 shows a plan view of the 380 L.

Figure 16-14: 380 L Plan View



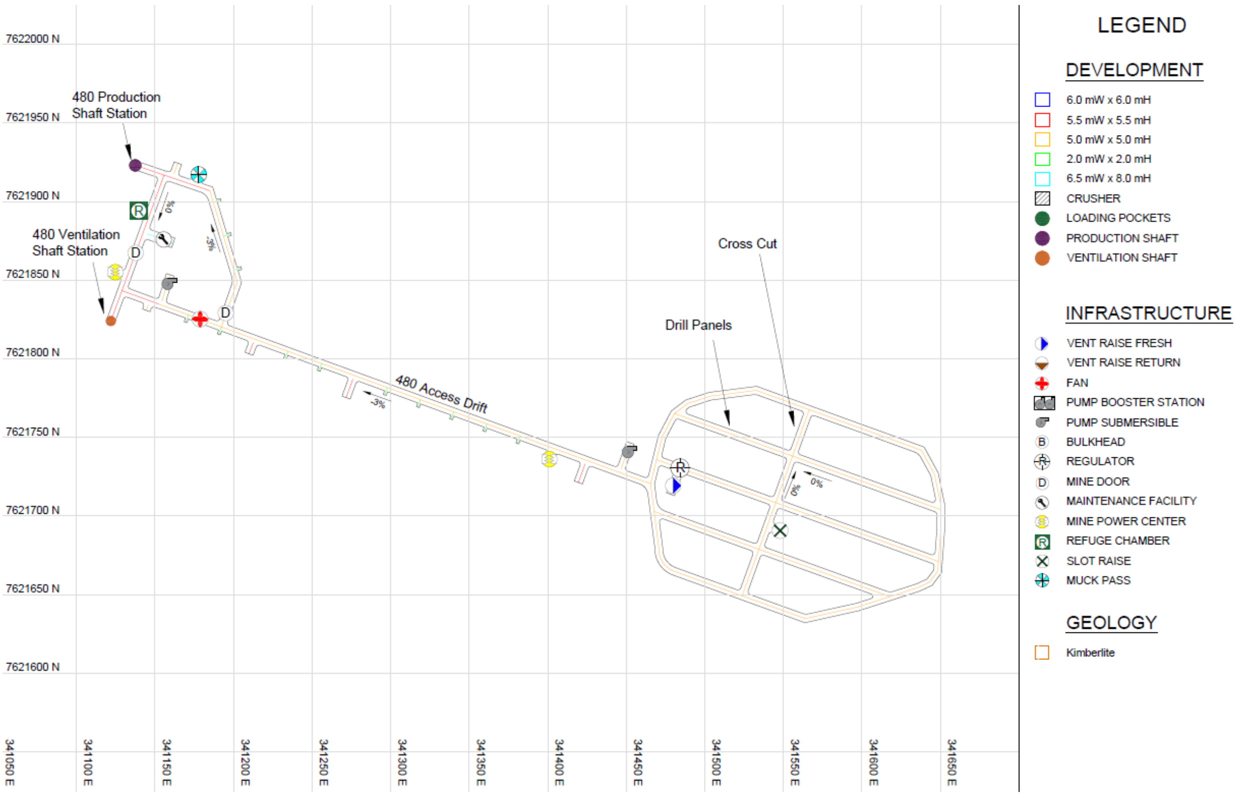
Source: JDS (2019)

16.7.5.3 480, 580, and 680 L Drilling Horizons

Three additional drilling horizons are located on at 480 masl, 580 masl and 680 masl. The 480 L and 680 L drilling horizons will be accessed directly from the shaft. The 580 L drill horizon will be accessed via a ramp driven within the kimberlite from 680 L to avoid poor host rock ground conditions expected between 480 L and 680 L. The access to the drill panels from the shaft stations by a single drive on the 480 L and 680 L drill horizons.

Drill panels and a crosscut will be driven in the same fashion as described in Chapter 16.7.5.2. Figure 16-15 illustrates the 480 L development. In this figure there are five parallel drill panels.

Figure 16-15: 480 Drill Horizon Plan View



Source: JDS (2019)

A slot raise, 3.0 m in diameter, will be driven from the 680 L drill horizon down to the 310 L extraction level. This will be driven in two segments by a raise bore machine. The first segment will be from the 480 L drill horizon to the extraction level, the second segment will be from the 680 L drill horizon to the 480 L drill horizon.

Fresh air will be supplied to the drill horizons by a FAR, 4.0 m in diameter, connected to surface. The FAR will connect to the 680 L, 580 L and 480 L drill horizons by access drifts, each equipped with regulators to control the ventilation airflow entering the level. Fans will be installed near the ventilation shaft on 480 L and 680 L to exhaust air up the ventilation shaft.

16.7.5.4 245 L Ventilation Level

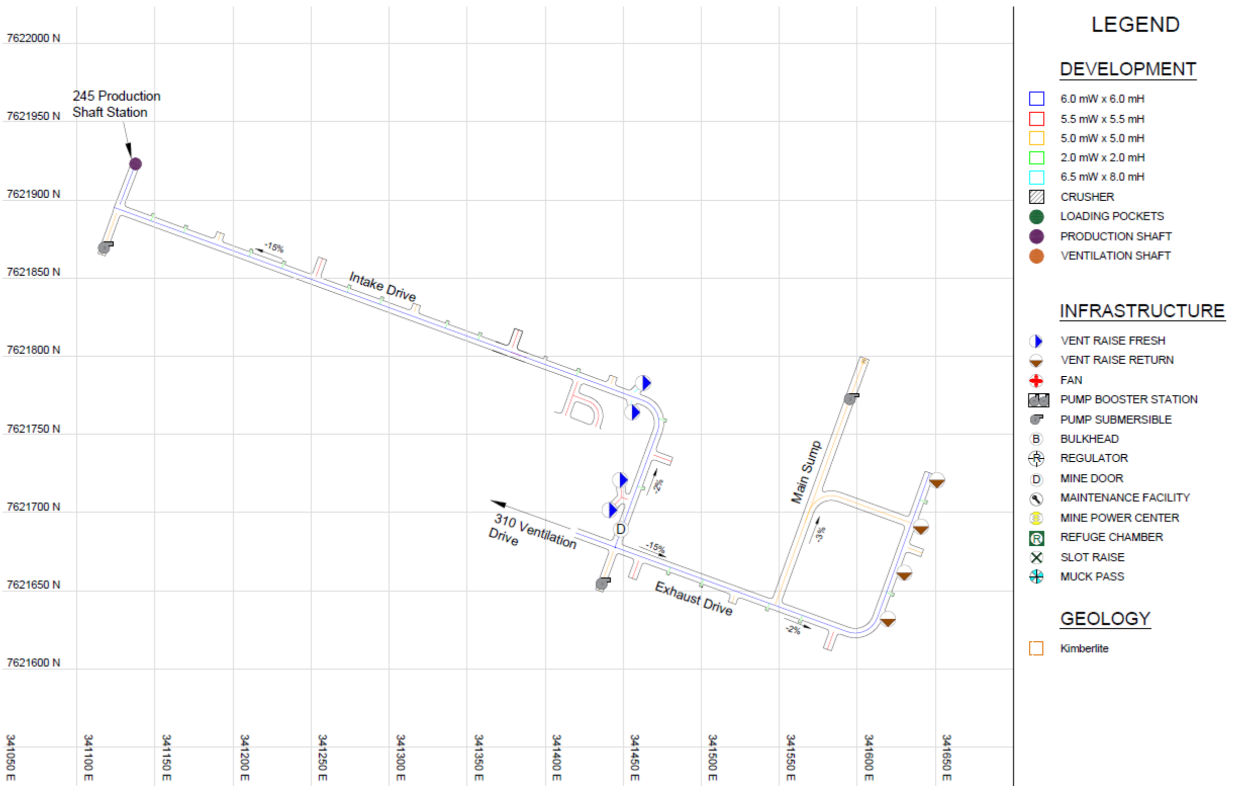
The ventilation level consists of a sublevel driven below the 310 L extraction area and a 15% incline which connects the P/S shaft bottom (245 masl) to the 310 L ventilation drive at approx. 292 masl. The ventilation level provides fresh air to the extraction level, the crusher room and the conveyor, and exhausts air from the extraction level and 380 L drill horizon.

The main fans are planned to be located on the 310 L ventilation drive as described in Chapter 16.8.2.

The main sump will be located at the low point of the exhaust drive and collects all water from the working areas. Water collected in the main sump will be pumped to the 310 L through a raise and directed to the 310 L pump station.

Figure 16-16 illustrates the 245 L development.

Figure 16-16: 245 Ventilation Level Plan View



Source: JDS (2019)

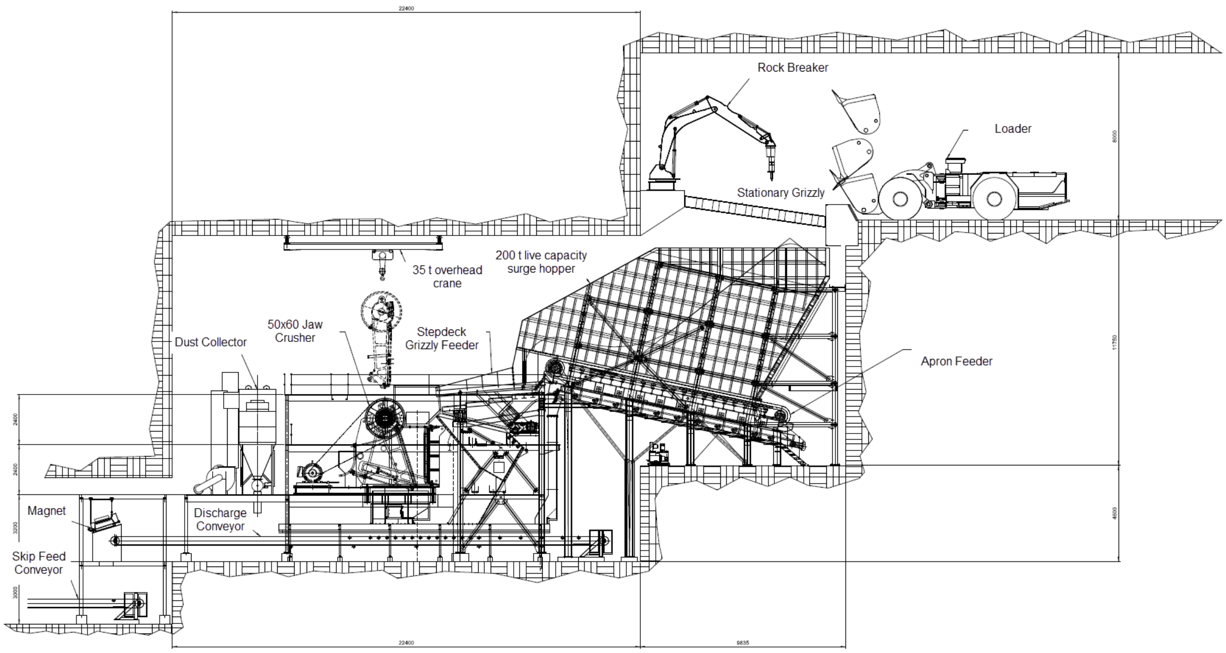
16.7.5.5 Crusher and Conveyor Levels

The crusher is located below the 310 L extraction area and will be accessed from the 310 L ventilation drive.

The final size of the crusher excavation will be 11 mW x 16.5 mH. A short raise will connect the extraction level to the crusher excavation, on top of which a fixed grizzly and rock breaker will be installed. The conveyor drive will accommodate a 1 mW conveyor and access for service equipment.

Fresh air is provided to the crusher and conveyor by the intake drift that is connected to the P/S shaft at 245 masl. Air is exhausted out of the conveyor to the 235 L ventilation shaft station.

Figure 16-17: Underground Crusher Layout



Source: Hatch (2019)

16.7.5.6 Loading Pockets

Two 40 m tall fine ore bins will store material prior to skip loading. Each bin will have approximately 3,500 t storage capacity.

The loading pocket bins will be accessed from the 310 L production drive and the 310 L ventilation drive. The 310 L production drive will provide access to the top of the loading pocket bins and the 310 L ventilation drive will provide access to the bottom of the loading pocket bins.

Loading pockets will be developed by a raise bore machine to drive a 3.0 m slot which will be slashed out to 7.0 m diameter using a long hole drill rig.

Conveyors will feed the loading pocket from the crusher as well as draw ore from the loading pockets for delivery to the skips. This is discussed further in Chapter 16.9.6.

A fan will be installed at the 335 L ventilation shaft station at the top of the loading pockets to exhaust air up the ventilation shaft and out of the mine. The bottom of the loading pocket bins will receive fresh air from the 285 L P/S shaft station and will be exhausted out by the 310 L ventilation drive. A door will be installed to control the airflow.

16.7.6 Stope Design

16.7.6.1 Stope Design Summary

Drill horizons are spaced at 100 m vertical intervals to accommodate the in-the-hole hammer (ITH) drill's effective drill length of a 150 mm (6") hole. Drilling of the stopes will be completed by mainly down holes on a 4.35 m burden by 5.00 m spacing ring pattern. The average length of hole per ring will be 58 m, with an average 34 t/m drilled. Stope production blasting will utilize a powder factor of 0.6 kg/t below the first drill horizon to ensure high rock fragmentation at the start of the shrinkage process. In the upper levels the powder factor will be reduced to 0.4 kg/t to match that of current open pit operations which produces excellent fragmentation.

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

During drill and blast the broken material will remain within the stope to provide wall support to the South Lobe. The swell created by blasting will be mucked from the drawpoints below the stopes to provide a blasting void.

Through areas of weaker host rock above the granite, a 15 m skin of kimberlite will be left temporarily around the walls of the lobe to prevent dilution and unraveling. This skin will be recovered later through drilling and blasting during final draw down of the muck pile.

16.7.6.2 Stope Design Criteria

Stopes were designed using Maptek© Vulcan 3D software and based on the following criteria.

Resource Geometry Limitations

The South Lobe is over 700 m in height and at the narrowest point is 100 m in diameter. The ore zone is continuous and lends itself to bulk stoping. The stopes are therefore not limited as much by the physical boundaries of ore and waste as they are by equipment capabilities and geotechnical requirements.

Sublevel Spacing

For long production holes it is common to use an in the hole (ITH) hammer long hole drill. The effective range of a Sandvik DU411 ITH drill equipped with 150 mm (6") bit is 100 m, which has been used to establish the sub level height of the stopes.

100 m tall stopes will be drilled in a downwards fan pattern with an average hole length of 58 m.

Crown Pillars and Sill Pillars

Crown and sill pillars are to be a minimum of 25 m. This criteria was determined through geotechnical modeling of the crown pillar stability during sequential bottom-up blasting of the South Lobe.

Pyramidal Blast Sequence

The large span of the South Lobe, particularly at higher elevations, may lead to unravelling of the back when undercut on a flat plane. This unravelling becomes stable as the back arches to form a dome shape. To prevent natural unravelling of the back the stopes have been designed to permit a stepped, or pyramidal, blasting sequence that will mimic and maintain a dome shape during production. A stope width and length of 31.5 m and 15 m respectively was selected to achieve the pyramidal blast sequence. Each stope will be 100 m tall and blasted sequentially in 17.5 m vertical increments until the final 30 m sill pillar is wrecked on retreat.

Protective Skin in Zones of Weakness

A stratigraphic unit comprised of mostly carbonaceous shale exists between the 480 L and 680 L shaft station. To prevent dilution and unraveling within the lobe during blasting a 15 m skin of kimberlite will be left temporarily around the walls of the lobe. This skin will be recovered later through drilling and blasting during final draw down of the muck pile.

Drill Pattern

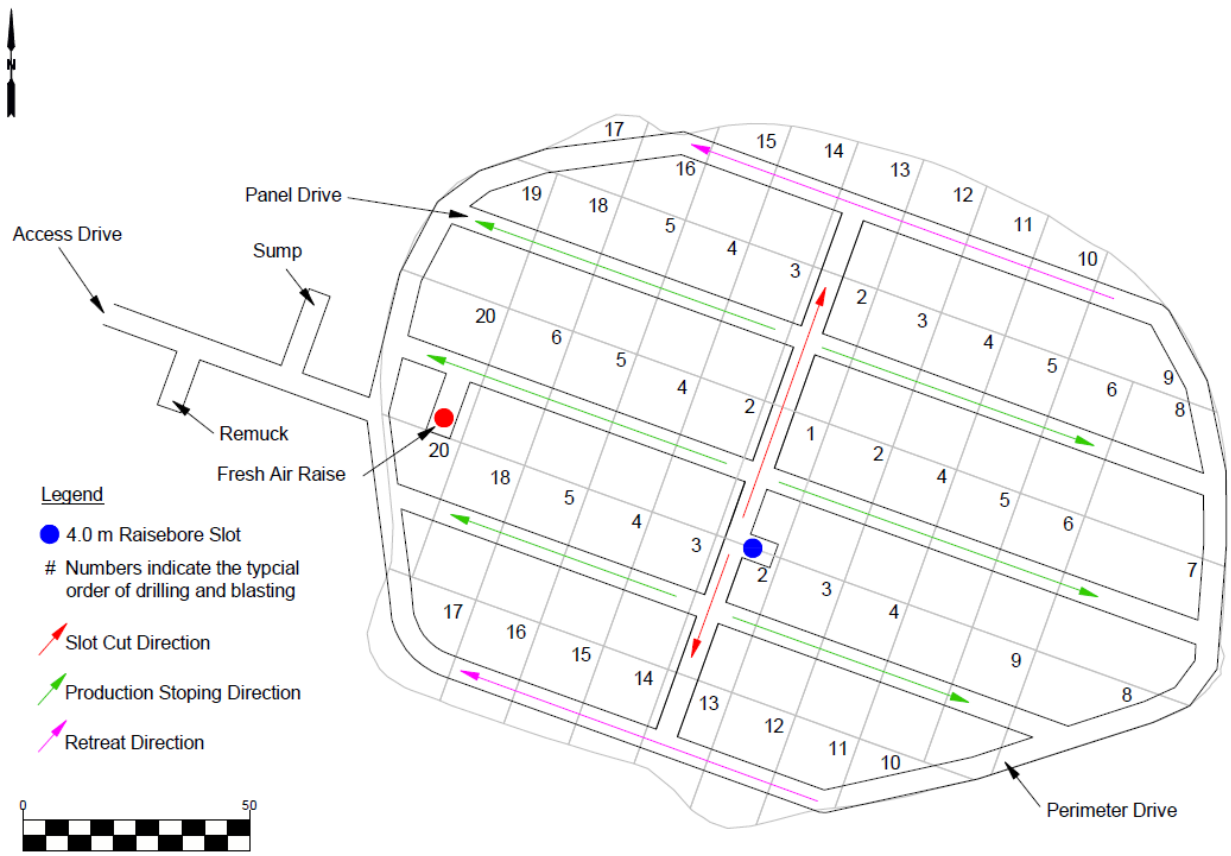
The open pit utilizes a 0.3 -0.4 kg/t powder factor and achieves excellent fragmentation. Underground stope drilling will be designed to achieve a similar powder factor with the use of 150 mm drill holes and a burden and spacing of 4.35 m and 5.00 m respectively. With these parameters the average length of hole per 100 m tall stope will be 58 m, with an average 34 t/m drilled.

Below the first drill horizon stope production blasting will utilize a powder factor of 0.6 kg/t to ensure high rock fragmentation at the start of the shrinkage process. This will be achieved by using the same burden and spacing but with a 165 mm (6.5”) drill bit.

16.7.6.3 Stope Sequencing

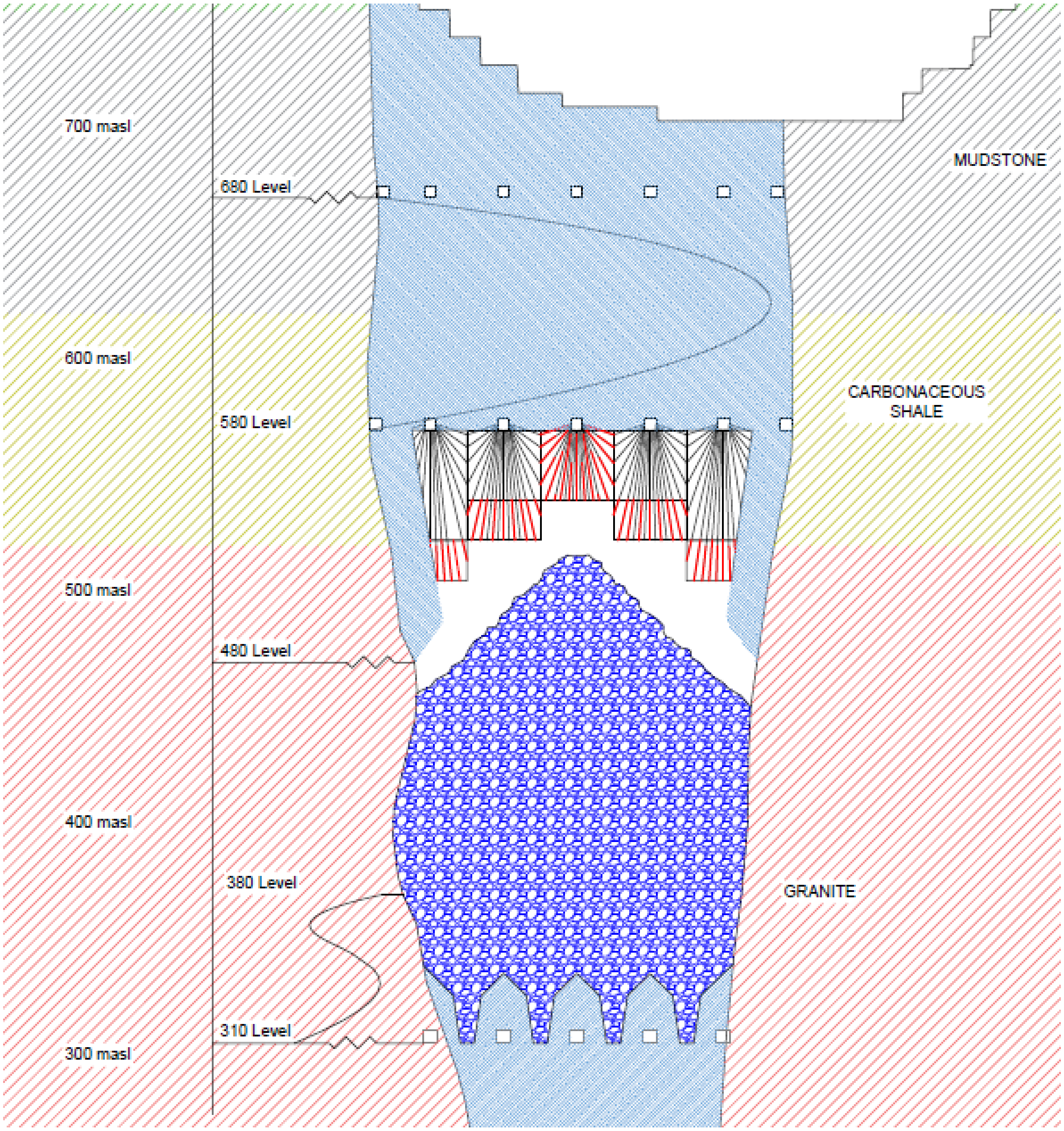
A slot raise will provide the initial blast void and free face for the long hole stopes to break into. A crosscut will be developed across the centre of the lobe, perpendicular to the direction of the drill panels on each drill horizon. A 3.0 m diameter raisebore will be driven vertically between these crosscuts and will be systematically slashed out using a long hole drill to provide a slot cut across the lobe. The slot will be stopped short of the perimeter drive on each horizon to provide man and equipment access to the back side of the drill panels. With the slot cut in place the long hole stopes will be drilled and blasted in retreat from the centre of the lobe, following a pyramidal blast sequence. Figure 16-18 illustrates in plan view the stoping sequence on a typical drill horizon. Figure 16-19 illustrates a cross section of the south lobe, showing the pyramidal advance of stopes while leaving a 15 m skin of kimberlite along the walls. In this figure the central stope is loading the final blast to wreck the sill pillar at that location.

Figure 16-18: Plan View of Typical Blasting Sequence



Source: JDS (2019)

Figure 16-19: Pyramidal Blast Sequence Schematic



Source: JDS (2019)

16.7.6.4 Design Optimization

Stopes have been largely designed around geotechnical constraints and the need to maintain a dome shape in the back while blasting. Should geotechnical conditions permit larger brows, or steps, between blasts there may be opportunity to increase stope dimensions in the X, Y, and Z direction to improve drill and blast efficiencies. The stope drilling and blasting design is very flexible and lends itself to optimization as the operation ramps up.

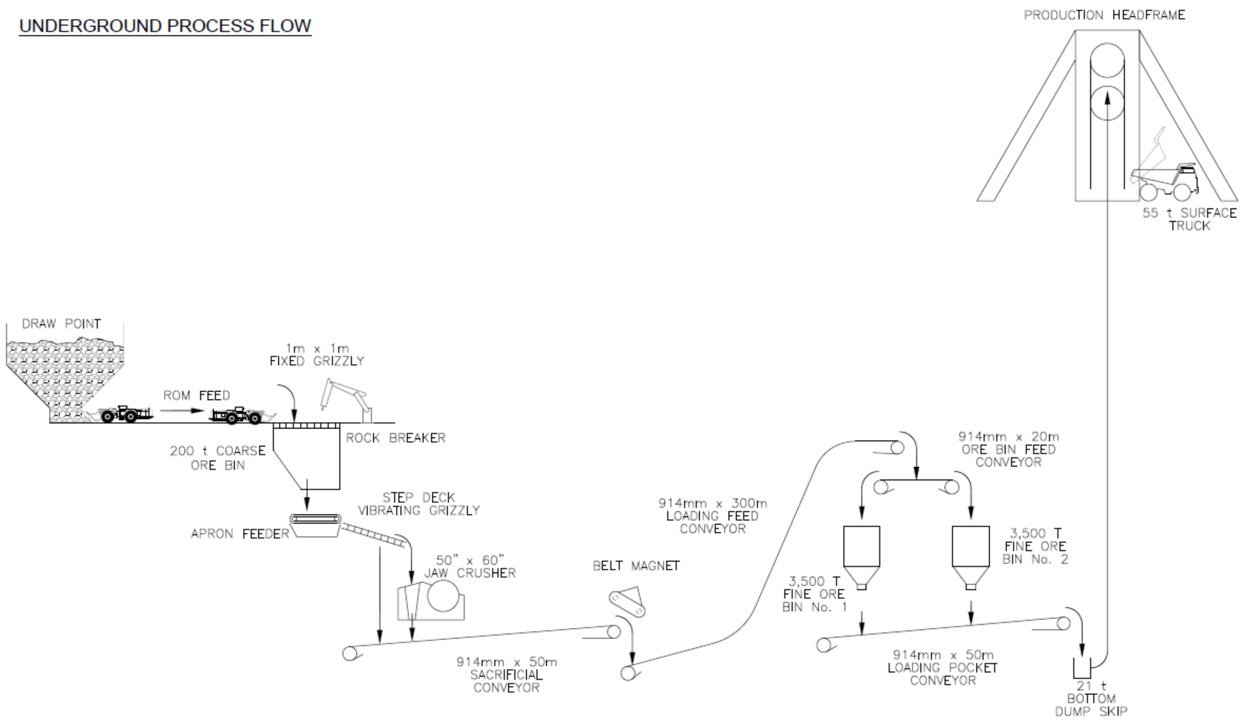
16.8 Mine Services

16.8.1 Comminution Circuit

The comminution circuit consists of single stage crushing and underground conveying to a double drum skip hoisting system. These systems are further described in Chapter 16.9.5.

Figure 16-20 illustrates the underground material flow from drawpoints to the surface.

Figure 16-20: Underground Material Flow Single Line Diagram



Source: JDS (2019)

16.8.2 Mine Ventilation

The ventilation network and fresh air supply quantities were designed to comply with South African ventilation standards. All work and equipment pertaining to mine ventilation facilities shall be designed, manufactured, installed and tested in accordance with the latest applicable local codes, regulations and standards. In the event of conflict, the more stringent standard shall apply.

16.8.2.1 Key Design Considerations

Mobile equipment is planned to be rubber-tired and equipped with Tier-2 diesel engines or above. The sulfur content in Botswana is 50 ppm, therefor Tier-4 engines are not applicable for this project. Airflow requirements for mobile equipment is to be the greater of:

- For CANMET certified engines, the CSA Ventilation Prescription for engines running on diesel with a sulfur content of 50 ppm; or
- 0.06 m^3/s/kW.

The design assumes that primary equipment such as loaders have an engine utilization of 100%, while auxiliary equipment will have a utilization of 25%.

The airflow required for mobile equipment and underground infrastructure is show in Table 16-8 and Table 16-9 respectively.

Table 16-8: Airflow Requirements for Underground Equipment

Equipment	Utilization	Power (kW)	Airflow Required (m^3/s)
2 Boom Jumbo	25%	110	3.30
LHD	100%	305	18.30
ITH Drill	100%	110	6.60
Bolter	25%	110	1.65
Shotcrete	25%	110	1.65
Transmixer	25%	190	2.85
Light Vehicle	25%	118	3.53
Grader	25%	108	6.48
Emulsion Charger	25%	110	1.65
Mobile Secondary Breaker	100%	110	6.60

Source: JDS (2019)

Table 16-9: Airflow Requirements for Underground Infrastructure

Infrastructure	Airflow Required (m^3/s)
Maintenance Shop and Lube Bay	40
Refuge Stations	10
Magazines	5
Loading Pocket Bins	15
Crusher	15
Main Conveyor	15
Refueling Station	10

Source: JDS (2019)

The following summarizes the maximum allowed velocity in all drifts based on industry standards:

- P/S shaft: 9.1 m/s;

- Ventilation Shaft: 16.3 m/s;
- Extraction Level and working areas of the mine: 5 m/s;
- Exhaust Drive (limited personnel): 6m/s; and
- Maintenance Shop and Explosives Magazine: 1 m/s.

16.8.2.2 General Arrangement

The proposed ventilation system consists of two networks providing separate air flows to the upper drilling horizons (480 L, 580 L and 680 L) and to the lower zone (380 L and below). An exhaust system is proposed with the main fans located underground, pushing air up the ventilation shaft and drawing fresh air down the P/S shaft and an in-pit ventilation raise. This will eliminate the requirement for an air lock at the shaft collar.

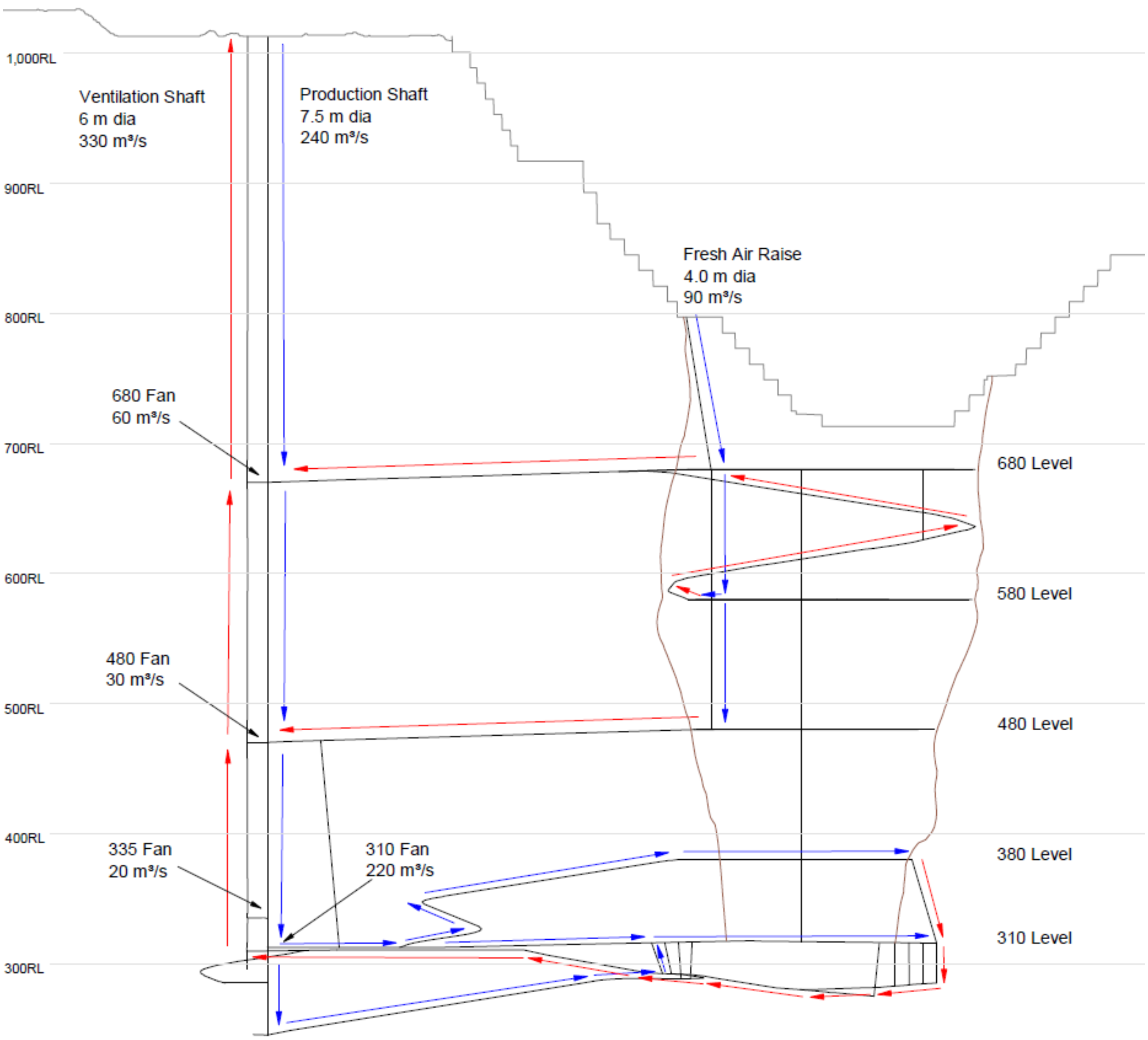
Parallel fans installed on the 310 L ventilation drift will draw fresh air to the lower zone. Fresh air will enter the area at both the 310 L extraction level and the 245 L ventilation level and will then be drawn into various locations within the mine. A fan installed on 335 L will control the airflow being pulled through the crusher and conveyor system.

Fans installed on the 480 L and 680 L will pull fresh air into the upper drilling horizons through a fresh air raise connected to surface. Regulators will be installed on each drilling horizon to ensure adequate airflow is pulled onto each level.

Development fans and ventilation ducting will direct fresh air to working areas during development until flow through connections are established, and permanently installed to supply fresh air through mine infrastructure that does not have flow-through ventilation.

Figure 16-21 illustrates the proposed ventilation network at KDM. Blue arrows indicate fresh air and red arrows indicate return air.

Figure 16-21: Proposed Ventilation Network



Source: JDS (2019)

Cooling cars with fans will be located at various locations throughout the mine to cool the air before it enters any working area.

16.8.2.3 Airflow and Fan Selection

The calculation of ventilation requirements for the mine was based on:

- Diesel equipment fleet and mining activity in work areas of the mine;

- Underground fixed facilities such as service bays, pump stations, etc.;
- Inactive areas that need nominal airflow to keep the temperatures within acceptable limits;
- Haulage routes of mobile equipment;
- Personnel working underground; and
- An estimated airflow leakage factor.

For sizing the underground infrastructure, peak ventilation demand was calculated followed by the airflow requirements at individual ventilation milestones. The following summarizes the airflow requirements:

- During peak production, 140 m^3/s is required to remove diesel emissions;
- 110 m^3/s is required to ventilate underground infrastructure;
- 40 m^3/s is required for haulage routes, worker comfort, air quality and network inefficiencies; and
- A 15% leakage factor has been assumed throughout the network.

The total designed ventilation capacity is 330 m3/s based on the equipment fleet profile, infrastructure requirement and crew allotment.

The main fan duty points during production were determined using Ventsim™ modeling software. The mine requires five fans during production. These fans will be commissioned underground.

For the main fans located on 310 L, fan selection considered parallel fan installations rather than one large fan for ease and flexibility of maintenance during operation, and for staging installations as airflow demand increases over time. Parallel fans are desirable to keep efficiencies high when ventilation requirements are low and only one fan is required, and to permit a reduced ventilation flow (as opposed to none) when fan maintenance is required.

The specifications for the main fans located underground are summarized in Table 16-10.

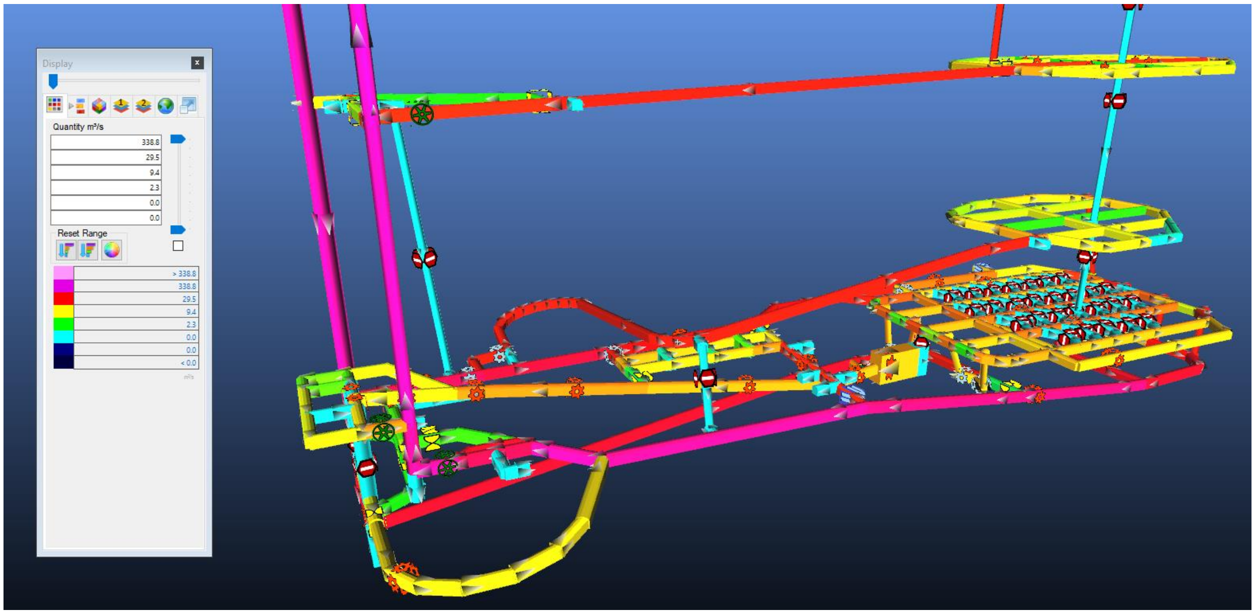
Table 16-10: Summary of Main Fan Duty Points

Location	No. of Fans	Quantity (m^3/sec)	Pressure (Pa)	Velocity (m/s)	Power (shaft kW)
310 Fan	2 – parallel	220	2,000	6.5	300
335 Fan	1	20	1,000	0.8	40
480 Fan	1	30	680	1.2	40
680 Fan	1	60	510	2.0	80

Source: JDS (2019)

Figure 16-22 shows an oblique view of the ventilation simulation during early mine production.

Figure 16-22: Oblique view of ventilation simulation



Source: JDS (2019)

16.8.2.4 Ventilation – Phases

Five ventilation milestones are identified in the life of KDM. They are:

- Shaft sinking;
- Early Pre-production;
- Pre-production;
- Drill and Blast Production Phase; and
- Mucking Production Phase.

16.8.2.4.1 Shaft Sinking

During shaft sinking, surface fans will be installed with ducting to bring fresh air to the working face. As shaft stations are constructed a crosscut will be driven between the shafts to establish a ventilation circuit. The shaft stations will connect at the 680 L, 480 L and 310 L.

16.8.2.4.2 Early Pre-production

During early pre-production, the ventilation network is limited, and all air will have to be ducted to the working face from the shaft. The airflow requirement during this phase is approximately 70 m³/s as there is limited headings available during this time. One jumbo will be developing from the P/S shaft and another jumbo will be developing from the ventilation shaft. Fresh air will enter from the P/S shaft and will return up the ventilation shaft. An exhaust fan will be installed and will be later replaced by the main fan when the airflow requirements are greater. Air will be directed to the face by 1.4 m fabric ducting and 75-110 kW fans.

16.8.2.4.3 Pre-production

A second crosscut will be driven between the production drive and ventilation drive near the maintenance shop, establishing a larger ventilation circuit and opening more available faces. Airflow requirements increase here to 165 m³/s. One out of the two 310 L main exhaust fans will be installed and be used during this phase. Air will be directed from the 310 production drift to the working face by 1.4 m fabric ducting and 75-110 kW fans. Curtain flaps will be utilized to direct the airflow to the working faces. The 335 L exhaust fan will be installed once the conveyor drift is connected to the 335 L ventilation shaft station to regulate airflow through the conveyor drift. The 480 L and 680 L exhaust fans will be installed once development commences on those levels.

16.8.2.4.4 Drill and Blast Production Phase

During the drilling and blasting production phase the second 310 L main fan will be installed. All development on the lower levels will be complete and only development on the 580 L and 680 L drill horizon levels remain. The airflow requirement for the mine at this stage will be approximately 330 m³/s. Raises between the 310 L production drive and the 245 L ventilation drive establish ventilation circuits at the extraction area and eliminate the need for development fans in the area.

16.8.2.4.5 Mucking Production Phase

When no more drilling and blasting is required, the airflow requirements will be approximately 240 m³/s. All development will be complete and only mucking of the material from the drawpoints remains. The 480 L and 680 L fans will no longer be in use as there will be no more activity on these levels. This phase will remain until the end of the mine life.

16.8.3 Mine Air Cooling

Due to the intake air conditions and high virgin rock temperatures (VRT), KDM UG will operate at elevated temperatures and it will be important to exhaust heat sources as quickly and efficiently as possible to minimize the risks associated with heat stress. Although there are no specific mine regulations in Botswana that dictate the need for mine air cooling, it is an international standard to achieve working temperatures below 27.5 degrees Celsius wet bulb (Twb) to maintain high levels of efficiency.

Where possible, temperature control mitigations have been exercised through mine design, ventilation controls, and mobile equipment selection. Enclosed cabs equipped with air conditioning will be utilized on mobile equipment where possible. Remaining heat loads have been addressed through the application of mine air cooling via underground refrigeration. It is estimated that mine air cooling will be required during the eight hottest months of the year.

KDM climate modeling was carried out using Ventsim™ software. Various heat loads occurring during production were input to the model to quantify the air refrigeration requirements for the mine.

16.8.3.1 Intake Conditions

The pressure, temperature, and humidity of the ambient air flowing into the mine will vary seasonally as well as day to night. These variances typically result in the transfer of heat to or from the intake shaft/raise walls and are damped by a thermal flywheel effect. Thus, the average temperature during the hottest months were taken as the basis for the estimation of refrigeration requirement for KDM. These are tabulated in Table 16-11 below.

Table 16-11: Average Summer Intake Conditions

Parameter	Value
Dry Bulb Temperature (T_{db})	32°C
Coincident Wet Bulb Temperature (T_{wb})	27°C
Relative Humidity (RH)	63%
Surface Barometric Pressure	102 kPa

Source: JDS (2019)

16.8.3.2 Geothermal Gradient and Rock Properties

The VRT at 310 L is estimated to be at 47°C. The mine geothermal gradient is 3.1°C per 100 m. This information is based on the site geophysical data interpretations of down hole surveys during hydrogeological studies. The geothermal gradient is typical for these parts in Botswana.

No rock geophysical properties were provided for the FS.

16.8.3.3 Maximum Reject Temperature

Wet bulb temperature index (WBGT) for heat stress indices was used to select a design parameter of 27.5°C wet bulb for the FS ventilation modeling. 27.5°C will ensure high efficiency of acclimatized workers. Workers may safely perform work underground up to 32°C wet bulb (Twb), albeit under short work durations and reduced efficiency. Work performed above 32°C wet bulb (Twb) must be planned on a case by case basis with application of appropriate heat stress safety measures.

16.8.3.4 Heat Loads

A total heat load of 5 MW is estimated to be imparted onto the ventilation system of KDM. The breakdown of heat loads is given in Table 16-12. Note that auto compression is included within the heat load simulations conducted by Ventsim and not tabulated here.

Table 16-12: Heat Load Distribution

Heat Source	Heat Load (kW)
Ground Water	510
Strata	1006
Diesel Equipment	2,045
Fans	604
Other Electrical Equipment	882
Total	**~5 MW**

Source: JDS (2019)

16.8.3.5 Cooling Design

Mine air cooling will supplement temperature controls with underground spot cooling equipment. Chilled water will be prepared underground by refrigeration machines (chillers) and pumped in an insulated closed-circuit network to mobile cooling coil air coolers (cooling cars) throughout the mine. Cooling cars will generate chilled air that is carried through the mine workings by way of ventilation regulators and auxiliary ventilation fans. The cool air will absorb heat produced by the mine and be exhausted to surface, effectively reducing the working temperature underground.

Cooling cars will be stationed near active working areas to combat localized heat sources associated with operating machinery. Some cooling cars will be permanently installed in strategic locations, while others may be relocated as the mine develops or local heat sources change locations.

To provide sufficient cooling for KDM, modular containerized reciprocating compressor water chillers are proposed on 310 L, 480 L and 680 L. These modular units contain the motor, compressor, and water pumps to and need only a water and power source for operation. The units are mobile by design and can be easily transported between working levels as required.

A total of 13 chillers are planned for KDM, two of which will be located on 680 L, one of 480 L, and the remaining 10 on 310 L where mine air and heat loads are the highest. At peak operation a total cooling load of 6.5 MWr will be employed. With a coefficient of performance (COP) of 3.5, a total of 1.9 MW electrical power is required to support this equipment.

Chillers will use Freon (R134A) to chill water supplied by several 10,000 L portable water containers stationed adjacent to the chillers. Chilled water will be pumped from the chillers to the cooling cars where the water runs through a series of baffles and finned tubing. 30 kW ventilation fans fitted to the cooling cars will force air through these fins which is chilled on contact, carrying the chilled air throughout the mine workings.

The water running through the cooling cars is heated by the air and this hot water is subsequently pumped to the nearest spray chamber for heat rejection from the mine. Spray chambers will be constructed on the 310 L, 480 L, and 680 L ventilation shaft stations and be comprised of a series of overhead spray bars. The heated water will be sprayed into the chamber and the exhaust ventilation will carry this heat up the ventilation shaft. As the water falls to the floor of the spray chamber it will be directed to a sump which will feed water either back to the chiller feed containers or to the main sump for ejection to surface.

A piping network will be installed to send water to and from the chillers to the cooling cars and be comprised of pipe dimeters ranging from 150 mm near 310 L chillers down to 50 mm as the network branches out to individual cooling cars. At peak operation a total of 260 L/s of chilled water will be pumped through the mine workings.

16.8.4 Water Supply

A single 100 mm diameter pipe will be installed in the P/S shaft to supply a maximum of 16.4 l/s of fresh water for use in the underground operation. A second line, 50 mm diameter, will also be installed in the P/S shaft to supply potable water.

16.8.5 Dewatering

16.8.5.1 Design Considerations

The mine has been designed with the following considerations:

- Ability to withstand a 1 in 100-300-year storm event, or approximately 300 m^3/hr;
- Capacity to manage underground dewatering activities up to 350 m^3/hr;
- Capacity to manage peak service water requirements of 50 m^3/hr;

- Storage availability for 18,000 m^3 of storm water during peak inrush; and
- Strategic placement of sumps and grading of underground development to minimize reliance on ditching.

Storm water modelling was conducted by Exigo. During such an event, approximately 40,000 m^3 of water may potentially report to the underground workings over a 96-hour period. Sumps and pump stations have been designed to manage this volume, as well as service water requirements and mine dewatering activities. It is likely that storm event will have little impact to the underground workings before the crown pillar is blasted and the muck pile is exposed to surface.

16.8.5.2 Dewatering System

Dewatering of KDM is through two 8 inch dirty-water pipelines installed in the P/S shaft between the 310L and 680L, and in the ventilation shaft between the 680 L and the shaft collar elevation. There will be a pump station located on 680 L and 310 L. On the 680 L, there will be a pumping capacity of 700 m^3/hr which is inclusive of ground water (350 m3/hr), service water (50 m3/hr) and a 100 plus year storm event (300 m3/hr). Five 375 kW pumps will be installed along with two 120 kW feed pumps.

On the 310 L extraction level, the pumping capacity will be 350 m^3/hr which is inclusive of service water (50 m3/hr) and a 100 plus year storm event (300 m3/hr). Three 375 kW pumps will be installed along with two 120 kW feed pumps.

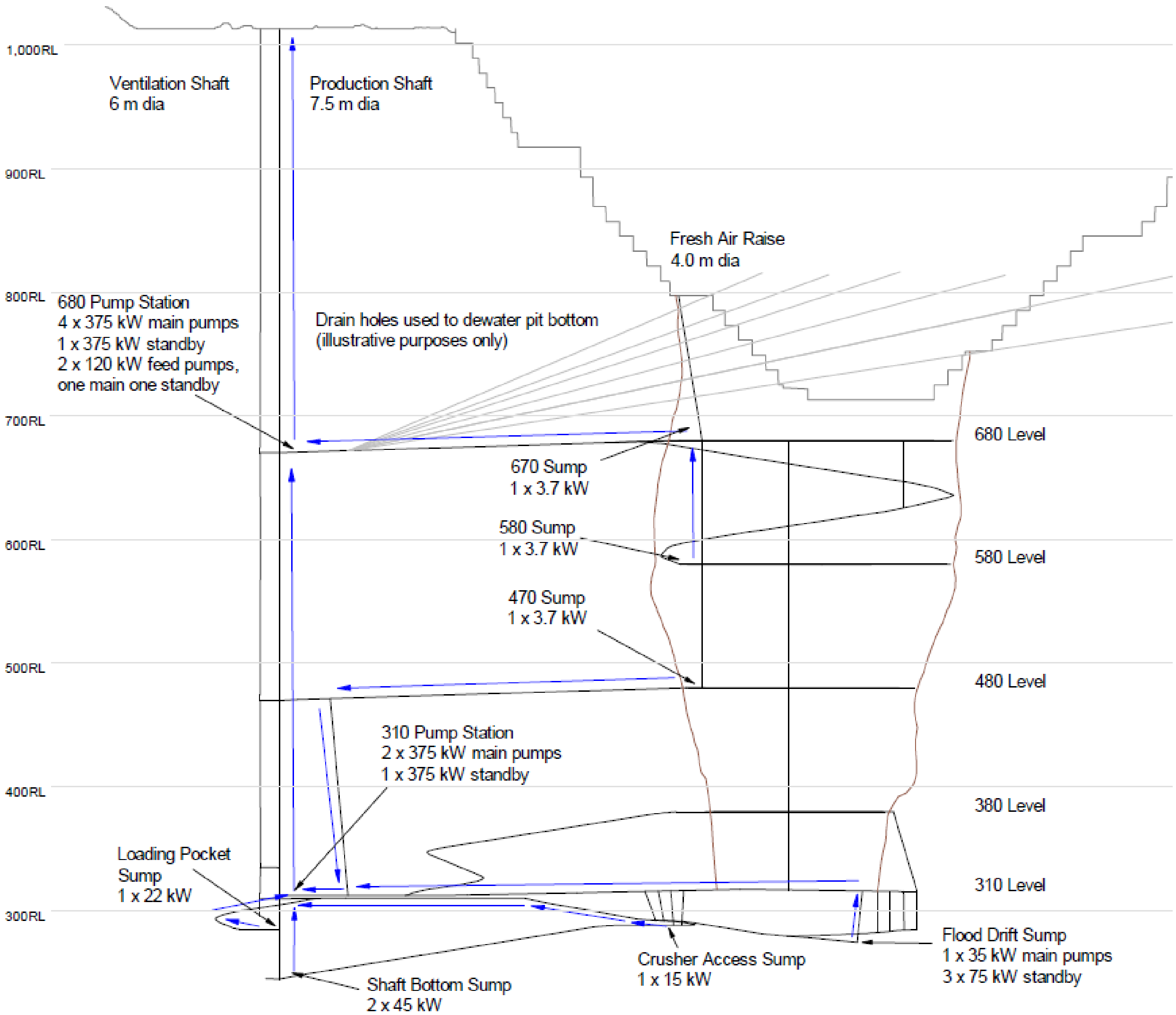
Sump stations are planned to be located throughout the mine. On the 680 L, 580 L and 480 L drilling horizons, one 3.7 kW sump pump will be installed to direct water to the pump stations. By the loading pocket, a sump station is installed with one 22 kW sump pump. The bottom of the P/S shaft will have two 45 kW sump pumps. One 15 kW sump pump will be installed by the crusher access.

In the flood drift sump, one 35 kW sump pump will be installed along with three 75 kW standby sump pumps which are designed to pump the anticipated flood water inflow up to the 310L pump station. The critical electrical and fixed infrastructure will be installed above the storm water flood level elevation to minimize the risk to this infrastructure.

Additional surface infrastructure has been designed to minimize ground water from entering the underground mine.

Figure 16-23 outlines the dewatering network at KDM.

Figure 16-23: Dewatering Network



Source: JDS (2019)

16.8.5.3 Water Disposal

From the underground operations water will be pumped to a settling pond on surface, which is then pumped into the existing dewatering ring which circles the open pit. From there the water either reports to the supply line or to the raw water tank at the process plant.

16.8.6 Electrical Distribution

The underground shaft area will be provided with two independent 11kV feeds from the main project substation to the shaft distribution switchgear.

Shaft distribution switchgear will be equipped with overcurrent protection devices. Horizontal shaft feeds, from the shaft distribution switchgear, will report to underground power distribution/motor control centers which will provide primary power supply to the mobile mine equipment.

Eight cables will feed distribution switchgears which will supply power to the following permanent mine power centers (MPCs):

- Underground crusher and conveyor loads;
- Shaft ventilation fans;
- Main pump stations on 680 L and 380 L;
- Submersible flood pumps; and
- Underground feed through two shafts.

The electrical system is designed with redundancy from the main project substation by bringing two 11 kV feeds to the underground area.

Each level within the mine will have a connection from the two underground feeds with permanent cables, feeding a loop around the perimeter drift on each level arranged by the ring main unit switchgear and feed through MPCs.

Each level has permanent distribution switchgear which allows the termination of incoming shaft cables and distribution of horizontal power feeds.

These distribution switchgears are to be installed at 680 L, 480 L, and 310 L from both the P/S shaft and the ventilation shaft.

MPCs will be installed at the major substations and near the south lobe to provide power to the fixed infrastructure and mobile equipment. Multiple mine load centers will be installed on each level to support mine development and production drilling and blasting on each level.

Multiple voltages will be provided to support the mining equipment, fixed equipment (pumps, primary ventilation fans and lighting), currently these voltages are based on the South Africa underground mines, however, they maybe opportunity to optimize the equipment voltages.

16.8.7 Mine Communications

An underground fibre network with wireless communications will be included. 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. The wireless system will be in place to facilitate an autonomous equipment operation should KDM choose to utilize the feature included in the specified equipment. A redundant leaky feeder system will be installed along the main drives on each level for emergency use.

16.8.8 Compressed Air

The compressed air system will support shaft sinking equipment during construction and mobile drill equipment during operations. Newer mining equipment often has built-in air compressors and does not need to be connected to the mine compressed air system. However, compressed air will be required by the ITH drills and the maintenance shops. Peak compressed air requirements are estimated at 3,290 cfm.

During preproduction construction three permanent 1,500 cfm compressors will be purchased with a fourth rental compressor of the same capacity. These four units will support the sinking of both shafts concurrently. At the end of shaft construction phase the rental compressor will be demobilized with the three permanent compressors remaining on site. Two of the permanent compressors will be operating during production with the third compressor on standby or to supplement the compressed air capacity during periods of peak demand. A total of 4,500 cfm of permanent compressed air system will be installed on surface and will be distributed underground. Compressed air lines will be installed in both the P/S and Ventilation shaft and branched off at each shaft station.

16.8.9 Explosives and Detonator Storage

There is currently a bulk explosives facility on site to service the open pit operations. This facility will be maintained to support the underground operations. Emulsion formulae for open pit and underground use is typically different, and therefore an additional emulsion tank may need to be installed (usually at the supplier's cost, built into the cost per kg supplied).

Bulk emulsion will be transported underground daily via the P/S shaft.

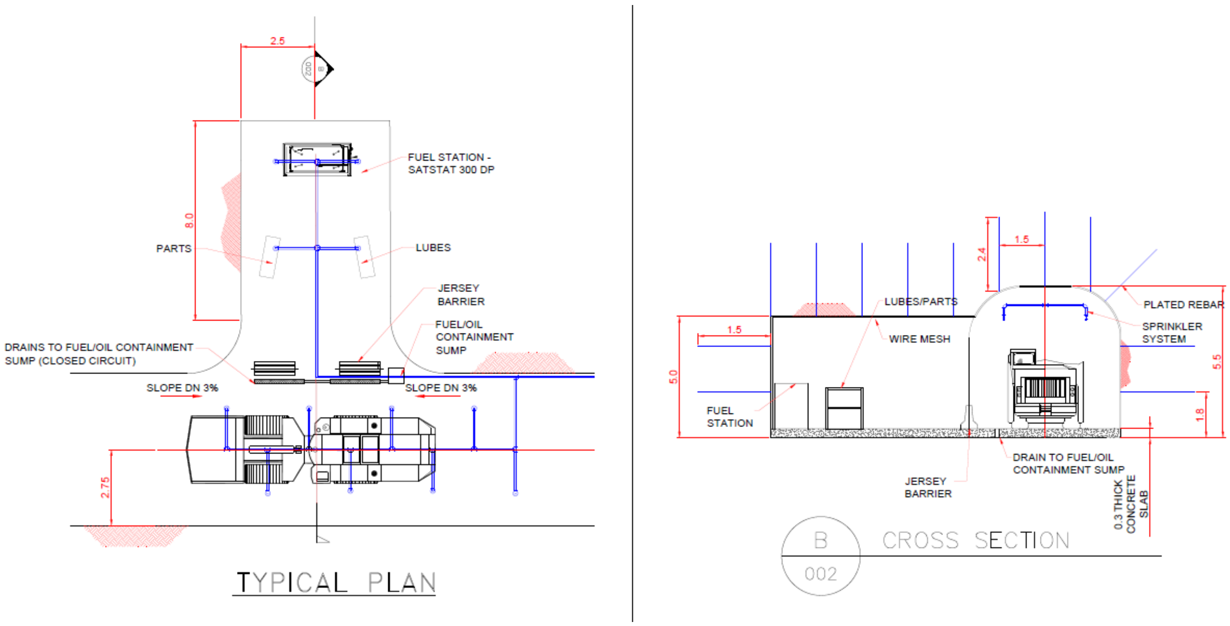
The existing surface magazines can accommodate the needs for underground operations. Underground explosive magazines will be located underground on 310 L, 480 L and 680 L and will contain enough storage to meet daily production.

16.8.10 Fuel Storage and Distribution

An equipment fueling and lube station will be located near the shafts on 310 L, 480 L and 680 L and will be able to provide fuel for the mobile underground equipment fleet. An additional fueling and lube station will also be located near the drawpoints on 310 L to provide quick access for the production LHDs. Fuel will be transported underground daily in portable containers and pumped into the fuel dispensing equipment. No fuel lines will be installed in the shaft or by borehole.

Figure 16-24 illustrates the type of fuel station that will be installed throughout the mine.

Figure 16-24: Fuel Bay General Arrangement



Source: JDS (2019)

16.8.11 Mobile Equipment Maintenance

The main underground maintenance facility will be constructed for services and repairs on 310 L. Access will be from the 310 L production drift and located in close proximity to the extraction area. The facility will be equipped with a wash bay, lube and oil change bays, electrical shop, tire storage, warehouse, and general service bays with 10 t bridge cranes.

The shop will be ventilated from 310 L production drive and will be connected to the exhaust drive for flow through ventilation. Fire doors will be installed to control ventilation during normal and emergency conditions.

Small maintenance facilities will be constructed on the 480 L and 680 L to service minor repairs.

A maintenance supervisor will provide a daily maintenance work schedule, ensuring the availability of spare parts and supplies, and providing management and supervision to maintenance crews. The supervisor will also provide training for the maintenance workforce.

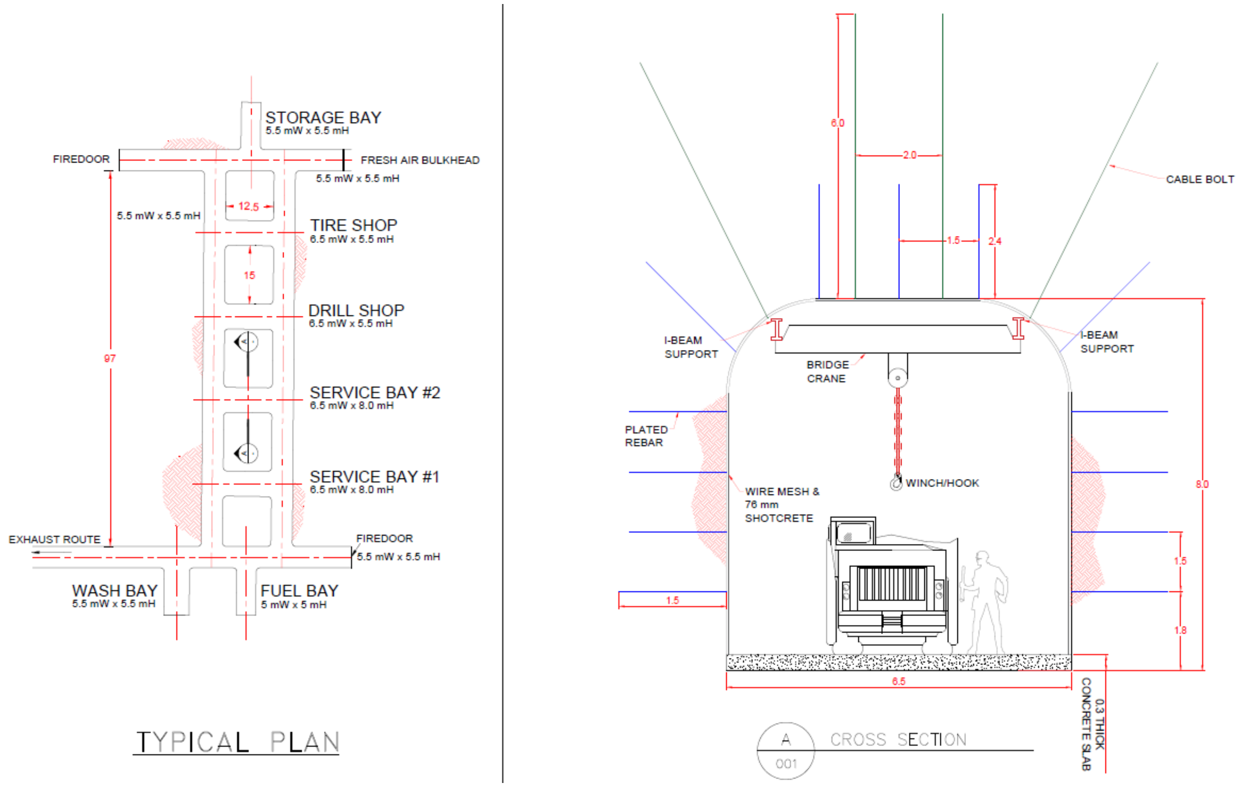
A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators will provide equipment inspections at the beginning of the shift and perform small maintenance and repairs as required.

During mine development all contractors will be responsible for mobile equipment maintenance and will have full access to the underground maintenance facilities. During commercial production maintenance will be performed by KDM employees. No marked contract for equipment maintenance is currently planned.

Figure 16-25 depicts the maintenance facility planned for the 310 production level.

Figure 16-25: Maintenance Facility General Arrangement



Source: JDS (2019)

16.8.12 Mine Safety

A permanent refuge station will be located on the 310 L and will also serve as a permanent lunchroom. Self-contained portable refuge stations will be located on the 480 L, 580 L and 680 L. The refuge chambers are designed to be equipped with dedicated fresh air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers doors are sealed to prevent the entry of gases.

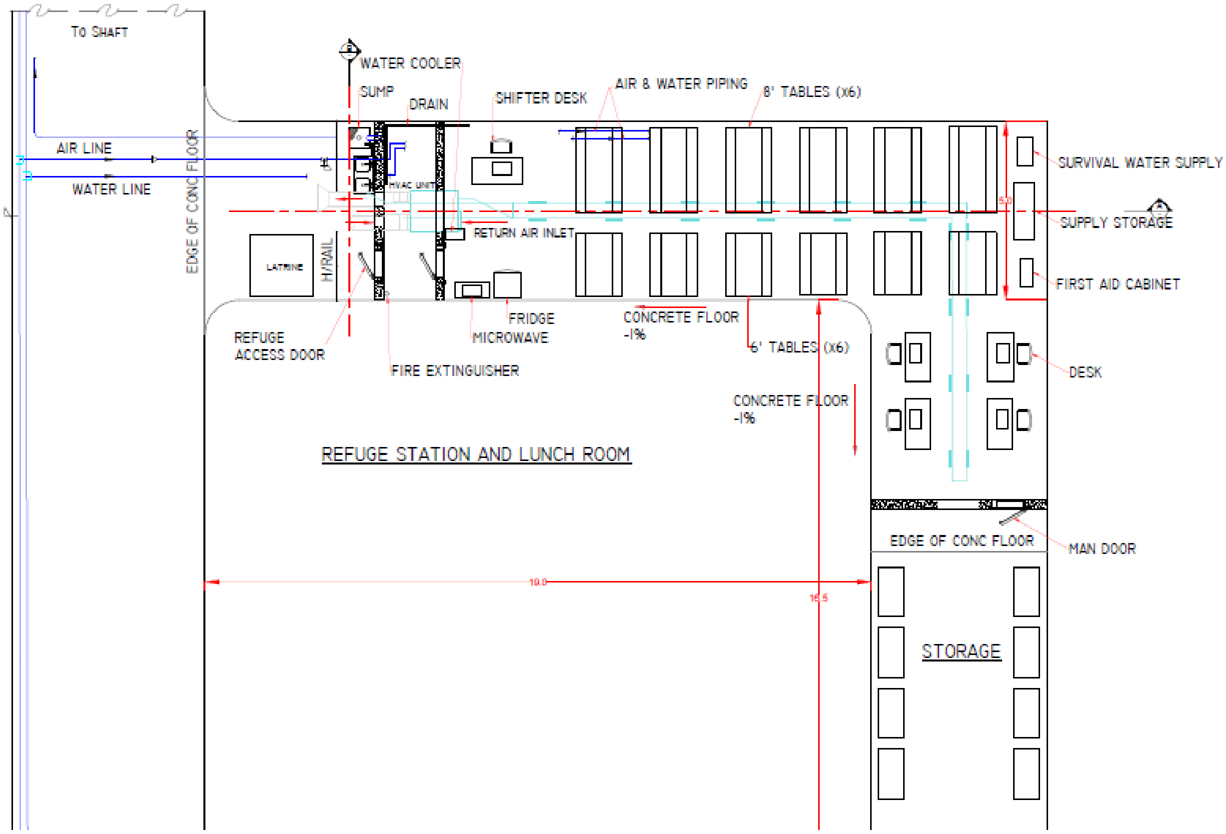
Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fueling stations, and other strategic areas. Every vehicle will carry at least one fire extinguisher of adequate size. All underground heavy equipment will be equipped with automatic fire suppression systems.

A fully equipped mine rescue team will be available every shift to respond to emergencies.

A stench gas system will be installed on the ventilation system and would be triggered to alert underground personnel in the event of an emergency.

Figure 16-26 represents the permanent refuge chamber and lunchroom designed for the 310 L.

Figure 16-26: Mine Refuge Chamber General Arrangement

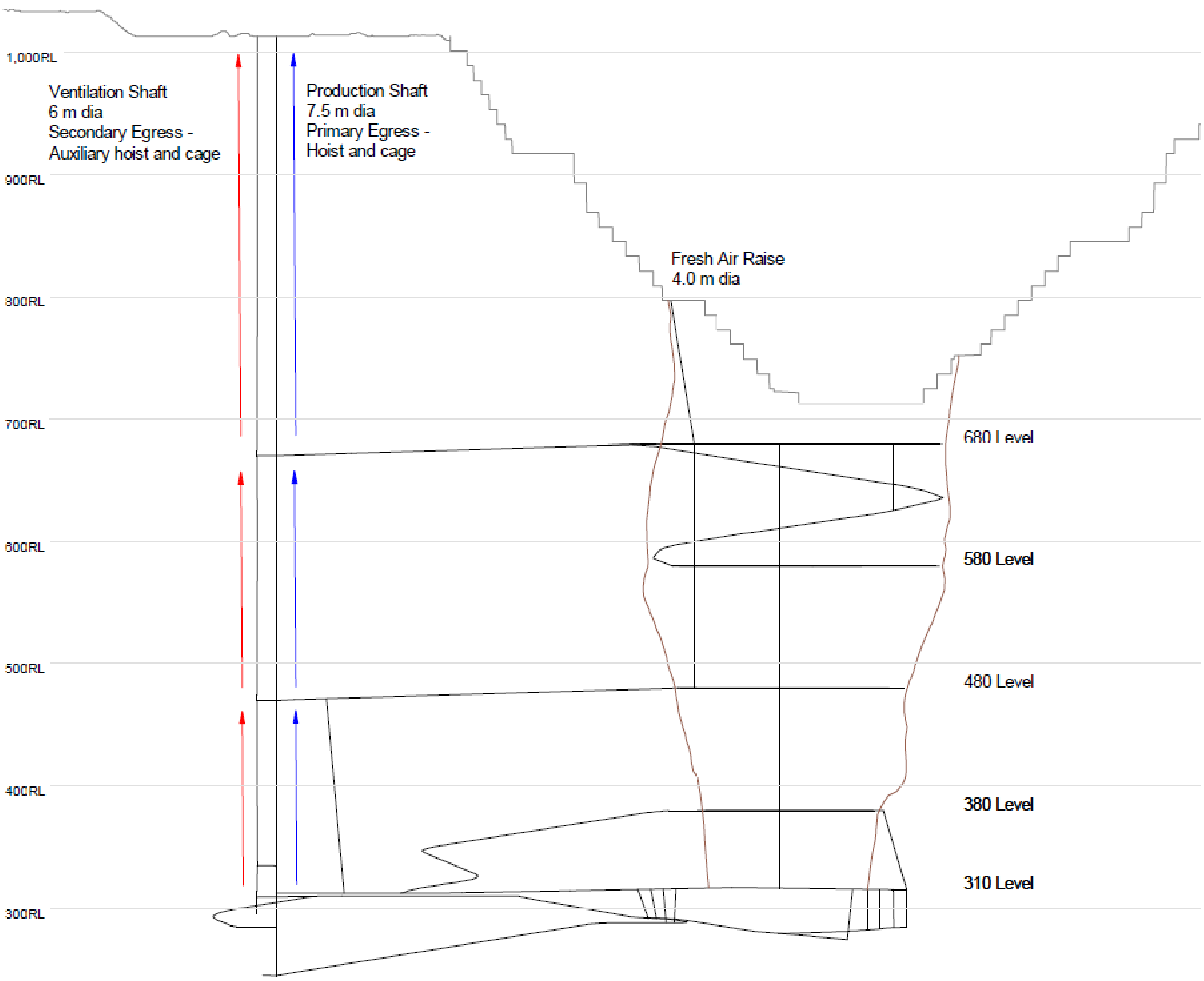


Source: JDS (2019)

16.8.12.1 Mine Egress

Primary mine access will be through the P/S shaft and will be equipped with a hoist and cage. Secondary emergency egress will be through ventilation shaft and will be equipped by an auxiliary hoist and cage powered by emergency generators.

Figure 16-27: Mine Egress General Arrangement



Source: JDS (2019)

16.9 Unit Operations

16.9.1 Drilling

Drilling activities will be undertaken by the following equipment:

- Twin boom jumbo; and
- In the hole hammer (ITH) longhole drill.

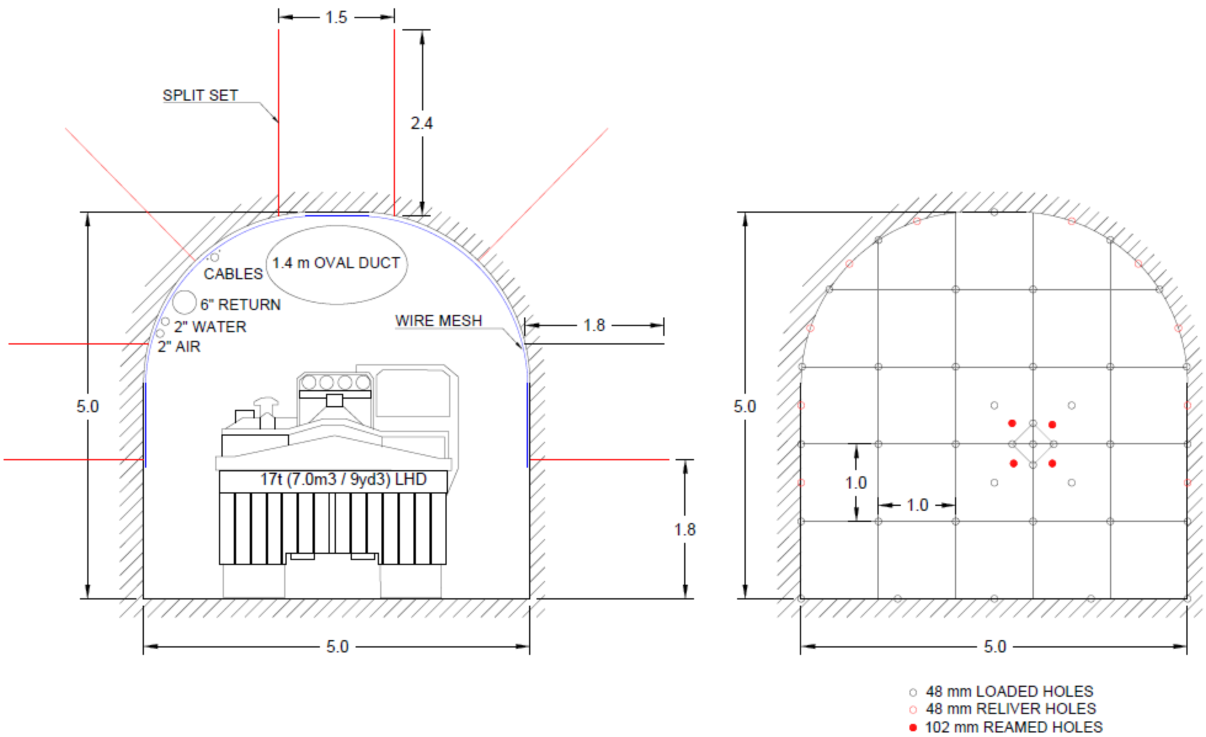
Drilling productivities (metre/percussion hour) were built up from first principles by drilling machine type and heading dimensions. Jumbo drilling rates average 75 m/hr in a 5.0 m x 5.0 m heading, and longhole drill machines average 12 m/hr or 105 m per shift.

16.9.1.1 Development Drilling

Development headings will be developed by two-boom electric jumbo drills. Jumbos will be equipped with 4.88 m (16”) drill steel and will advance 4.4 m per blast. Jumbo advance is budgeted to an average of 3.5 m/d per machine in priority headings and 2.5 m/d per machine in non-priority headings, to a maximum 11 metres per day per machine over four active faces. This equates to approximately 2.25 rounds per day per machine when four faces are available.

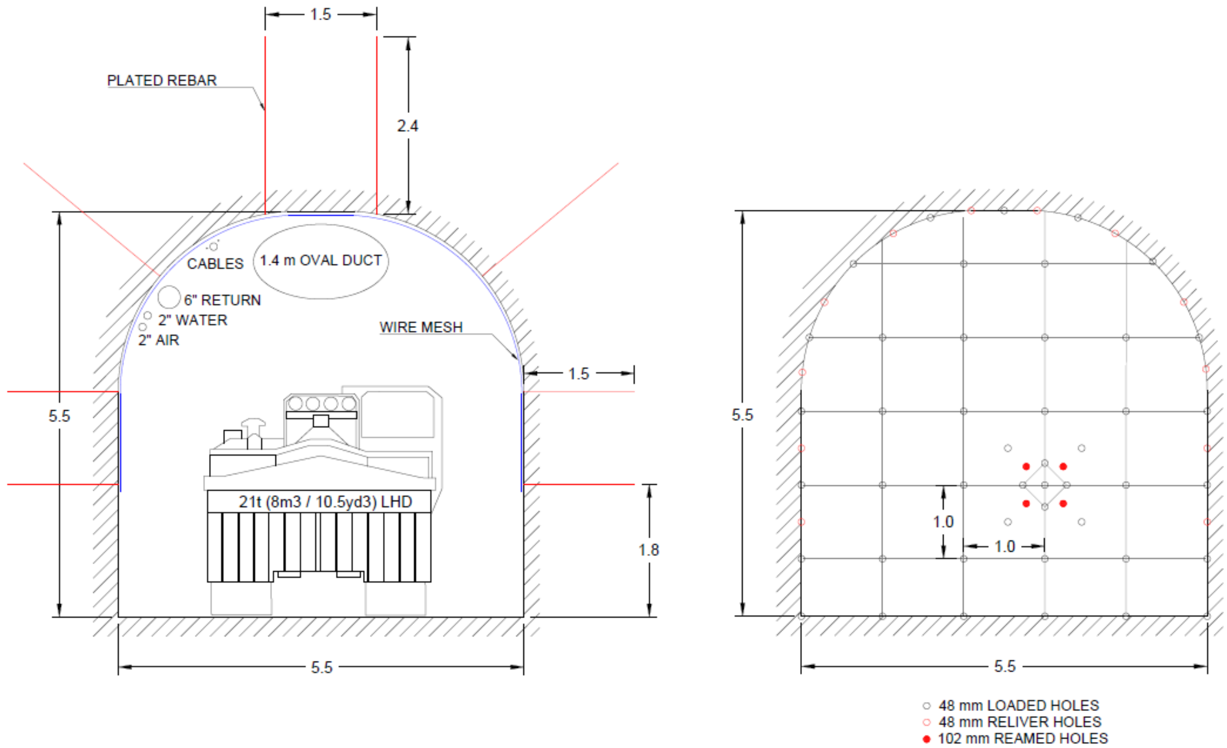
Typical jumbo drill patterns are depicted in Figure 16-28 through Figure 16-30.

Figure 16-28: Development Cross Section for Typical 5.0 m x 5.0 m heading



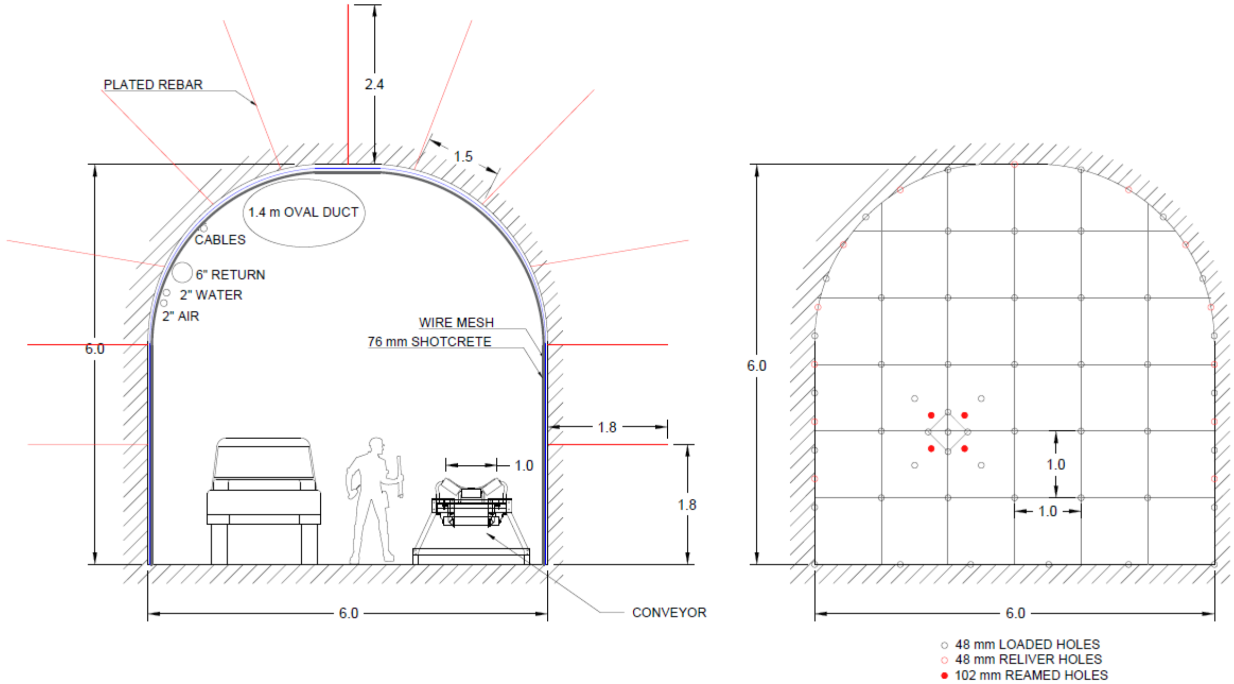
Source: JDS (2019)

Figure 16-29: Development Cross Section for Typical 5.5 m x 5.5 m heading



Source: JDS (2019)

Figure 16-30: Development Cross Section for Typical 6.0 m x 6.0 m heading



Source: JDS (2019)

16.9.1.2 Production Drilling

Longhole production drilling will start with 45 m downholes drilled from the 380 level to the top of the drawbells. 165 mm diameter holes drilled on a 4.35 m burden and 5.00 m spacing will yield an average powder factor of 0.6 kg per tonne. This relatively short sub level with relatively high powder factor has been designed specifically to ensure high drill accuracy and high blast fragmentation to initiate the shrinkage operation.

Above the 380 L, sublevels are increased to 100 m vertical spacing. Longhole drilling of mainly down holes with 150 mm diameter is planned on a 4.35 m burden and 5.00 m spacing to yield an average powder factor of 0.4 kg per tonne. This material will experience more comminution within the pipe as muck is pulled from the drawbells, so a lower powder factor will be used. The open pit operations currently drill and blast ore to a powder factor of approximately 0.4 kg per tonne.

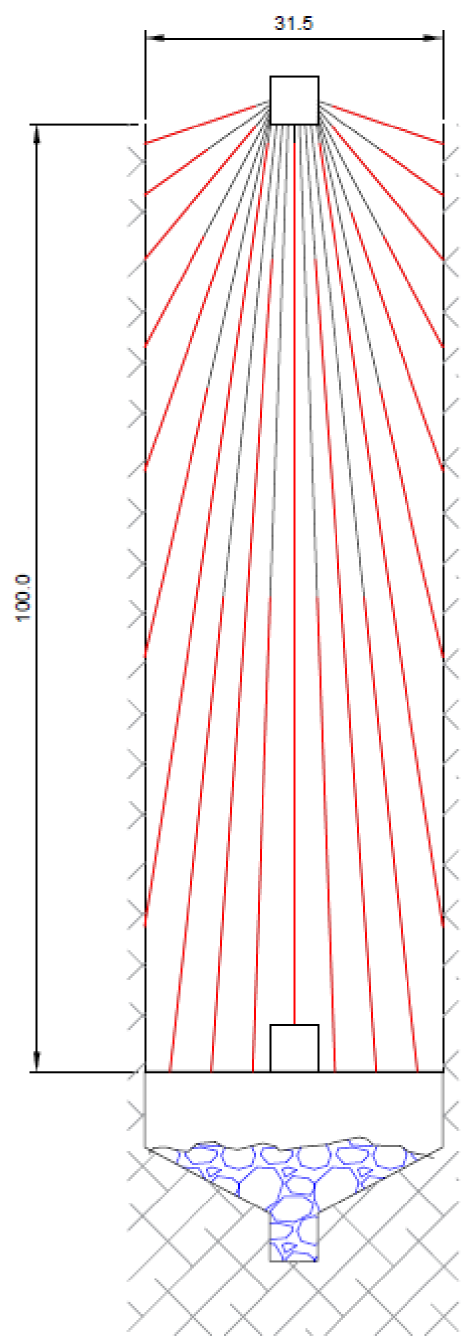
Some stoping would include drilling of upholes, particularly in the crown pillar, with a maximum length of 30 m to ensure emulsion can be held in the hole.

The average drill length for a typical 100 m tall ring pattern is 58 m and yields 33.9 t per metre drilled including a 10% redrill factor. Figure 16-31 depicts a typical ring design.

Figure 16-31: Long Hole Stope Ring Design

Drilling 380+	Units	Drilled (m)	Total Loaded (m)
Hole 1	m	14	12.26
Hole 2	m	16.31	8.15
Hole 3	m	20.7	18.24
Hole 4	m	28.01	13.24
Hole 5	m	39.67	28.85
Hole 6	m	58.16	29.08
Hole 7	m	85.7	83.68
Hole 8	m	100.63	50.32
Hole 9	m	100.28	85.99
Hole 10	m	100.07	50
Hole 11	m	100	98
Hole 12	m	100.07	50.00
Hole 13	m	100.28	85.99
Hole 14	m	100.63	50.32
Hole 15	m	85.7	83.68
Hole 16	m	58.16	29.08
Hole 17	m	39.67	28.85
Hole 18	m	28.01	13.24
Hole 19	m	20.7	18.24
Hole 20	m	16.31	8.15
Hole 21	m	14	12.26
Average Hole Length	58 m		

Drill Hole Diameter	mm	150
Burden (19 x dia)	m	4.35
Spacing (1.25 x B)	m	5.00
Stope Dimensions:		
Stope Height	m	100
Stope Width	m	32
Ore Density	t/m3	2.9
Blasted Tonnes	t	45,698
Drilled Holes per Ring	#	21.00
Drilled Metres per Ring	m	1,227
Redrill	%	0.10
Redrill Metres per Ring	m	123
Tonnes per metre drilled	t/m	33.9
Explosives	g/m3	1,200
Column Load	kg/m	21
Total Load	kg	18,186
Powder Factor	kg/t	0.40



Total Loaded Pattern

Source: JDS (2019)

16.9.2 Blasting

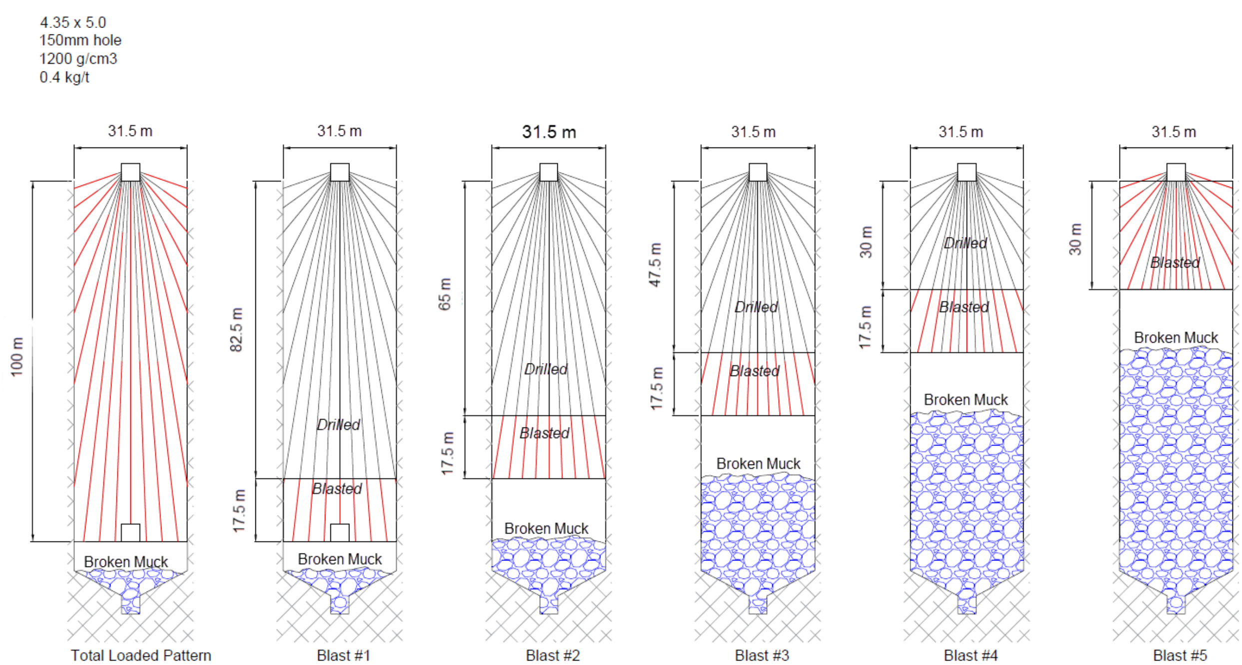
For explosives use, blasting crews will be trained and certified. Bulk emulsion will be used for production blasting and development rounds. Boosters, primers, detonators, detonation cord and other ancillary blasting supplies will also be utilized. Smooth blasting techniques may be used as required in headings, with the use of trim powder for loading the perimeter holes.

Bulk explosives will be manufactured on surface in accordance with current Botswana Explosives Regulations. The blasting crews will pick up the estimated quantities of explosives required for each shift using explosives cartridges and transport vehicles and deliver those explosives to working faces and explosives-loading equipment underground. Excess explosives and accessories will be returned to the secure powder magazine every shift. All explosives and detonators in and out of the magazines will be documented as per Botswana Explosives Regulations.

During the pre-production period, blasting in the development headings will be done at any time during the shift when the face is loaded and ready to blast provided all personnel underground are in a designated Safe Work Area and ventilation is adequate. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of each shift. Where ventilation allows, multi-blasting of isolated high priority development headings is possible.

Each 100 m tall stope will be blasted in several vertical segments, maintaining a minimum 30 m sill pillar below the drill panel until the final blast is taken and access to the drill panel is lost. Figure 16-32 illustrates the drill and blast sequence of a single stope.

Figure 16-32: Stope Blast Sequence



Source: JDS (2019)

Stopes will be blasted such that a dome shape is created across the South Lobe. This is to promote geotechnical stability within the lobe and prevent slabbing of large blocks into the muck pile. Figure 16-19 (see chapter 16.7.6) depicts a cross section of the South Lobe during drill and blast. In this figure five stopes that have been drilled (black) and are loaded (red) in preparation of the next blast.

16.9.3 Ground Support

Ground support will vary depending on the size of opening, service life, and ground conditions. Table 16-13 outlines the different ground support applications planned for KDM UG.

Table 16-13: Ground Support Regime

Support	Description
Temporary Support (ore)	Bolt and Welled Mesh 2.4 m backs & 1.8 m walls down to 1.8m grade line above the floor 1.5 by 1.5 pattern (split set)
Permanent Support (waste)	Bolt and Welled Mesh 2.4 m backs & 1.8 m walls down to 1.8m grade line above the floor 1.5 by 1.5 pattern (rebar)
Shotcrete	7.6 cm (3") To be applied to all of the extraction area and maintenance facility
Cable Bolting	At all intersections, 6.0 m cables to be installed on a 2.5 m x 2.5 m pattern
Drawpoints Additional Support	Two steel arches bolted and concreted in, set back from the brow.
	Nose pillars to receive steel plate 1.5 m from the ground wrapped around nose of the herringbone pillar; post bolted with 6m cables (twin-strand)

Source: JDS (2019)

Ground support will be installed in accordance with specifications based on geotechnical analysis for the various rock qualities expected. The massive (unstructured) nature of the of the kimberlite and granite renders the ground support design inapplicable to empirical systems such as RMR, Mathew's Q or modified Q. These systems rely on block size, jointing, water flow and joint condition, which are not applicable to unjointed rock masses. The ground support design has, therefore, been based on industry standards for life of the opening and function of the excavation. The proposed ground support has been evaluated by Itasca using Flac 3D to confirm suitability of the design during the various phases of the mine life. The proposed ground support was deemed suitable with the pyramidal opening sequence.

Primary ground support will be installed post-mucking of the blasted drift. No additional development will be commenced in the heading prior to the installation of primary ground support. At no time will mine workers be under unsupported ground. Secondary and tertiary support may be installed out of the development cycle by the service crew in accordance with the ground support management plan (to be further developed during detailed design).

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with different stratigraphic units within the waste rock. Discretion will be made by the development lead as to which ground support is required, with additional review and recommendations provided by the on-site geotechnical engineer.

Electric-hydraulic bolters and shotcrete spraying machines will be used. Shotcrete will be applied when required as a wet mix, which is mixed in a transmixer and pumped into a skid mounted shotcrete sprayer.

Regular pull tests will be conducted on-site to ensure adequate installation of resin rebar, split set, and cables bolts are being done. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy in accordance with the ground support management plan and QA/QC.

16.9.4 Mucking

The LHD selected for development mucking has a 17 t (7 m^3) nominal capacity. For development, LHD's will typically muck a blasted round to a nearby re-muck bay in order to clear the working face prior to ground support installation. Rock temporarily stored in the re-muck is then either trammed to a rock pass or loaded into a haul truck.

There will be 54 drawpoints over five extraction drives in operation throughout the life of mine. Material will be systematically mucked from the drawpoints by three LHDs to maintain the desired muck pile shape within the lobe. During drill and blast operations this shape will be a cone to mimic the dome shape created by the blast sequence. During final draw down the muck pile shape will be an inverted cone to maximize wall support until the lobe has been emptied.

Stope ore will be mucked with a 21 t (11 m^3) LHD and trammed directly to the crusher coarse ore bin grizzly. In the event the crusher cannot accept ore feed, either for capacity or maintenance reasons, the LHD will muck into one of several remuck bays located adjacent to the grizzly and later rehandled when space becomes available.

LHD cycle times and quantity requirements were calculated from first principals. An average haul distance of 160 m was used for the tram distance from the drawpoints to the grizzly. Other LHD operating parameters are shown in Table 16-14. Both the 17 t and 21 t LHDs are limited by bucket capacity rather than operating load.

Table 16-14: LHD Operating Parameters

LHD Operating Parameters	Units	21 t LHD
Tramming Capacity	t	21
S.G. Bulk	t/m^3	1.89
Target Fill Factor	%	95%
Target Bucket Size	m^3	11.1
Largest Available Bucket	m^3	10.7
Selected Bucket	m^3	10.7
LHD Capacity Actual	t	19.2
LHD Capacity Actual	m^3	10.2
LHD Loaded Tram Speed	km/hr	5
LHD Empty Tram Speed	km/hr	10
Operator Efficiency	%	90%
Load	min	0.50
Dump	min	0.25
Maneuver	min	0.25

LHD Operating Parameters	Units	21 t LHD
Mucking Fixed Time	min	1.11

Source: JDS (2019)

Three production LHDs will be required to meet the target production rate. This has been calculated based on number of loads, cycle times, and available working hours per day. An Arena simulation was prepared to test the impact of LHD requirements during events of unscheduled maintenance and longer than average tram distances during periods of drawpoint rehabilitation. This simulation also concluded that three production LHDs would be required to meet production. The Arena simulation is discussed in more detail in Chapter 16.7.3.1. Development LHDs will be available on standby to assist with production mucking if required.

LHDs will be inspected before each shift and returned to the maintenance facility at end of shift for fueling, lubrication, and preventative maintenance (PM) if required. LHDs are expected to require refueling every seven operating hours and will report to the fuel station some 200 metres from the working area.

Diesel fired LHDs have been selected for all mucking activities at KDM.

16.9.5 Crushing and Conveyance

LHDs will tram ore from the drawpoints directly to a single stage crushing plant. The crusher will process 450 t/h or 7,200 t/d of material, operate 16 hours per day based on a utilization of 65% and produce a final product P_{80} of 150 mm.

Material will be dumped onto a 1,000 mm static grizzly above the crusher dump pocket. The material will discharge through the static grizzly into the 200-t crusher feed hopper. Oversized material from the static grizzly will be size reduced using a rock breaker mounted beside the static grizzly.

An apron feeder will draw material from the dump pocket to feed the vibrating grizzly feeder at a rate of 450 t/h. The vibrating grizzly oversized material will feed directly into a 1,270 mm x 1,524 mm (50” x 60”) jaw crusher with an installed power of 250 kW. The undersized -120 mm material will bypass the crusher and feed directly onto the crusher discharge conveyor. The primary crushing stage will produce a product P_{80} of approximately 150 mm and an F_{100} of 228 mm at a crusher closed side setting (CSS) of 152 mm.

The crusher discharge conveyor will pass through a 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 crusher discharge conveyor will feed material onto the skip feed conveyor for transport to 335 L. The skip feed conveyor discharges onto the skip reversible transfer conveyor which feeds one of two crushed ore storage bins, each with a capacity of 3,500 t.

The crushing area is equipped with a 35-t crane for maintenance, compressed air, dust collection and a self-cleaning belt magnet.

16.9.6 Hoisting

The loading pocket bins feed a skip loading conveyor, where material is dropped into one of two 21 t loading flasks which in term feed the 21t bottom dump skips. Skips will be hoisted opposing to one another (when one is going up, the other is going down) on two-minute skip hoisting cycles. The average electrical power load for the rock hoisting cycle is 3,570 kW (RMS). The rock hoisting capacity is 3.2 to 3.5 Mt/a based on an annual average availability/utilization of 65 to 70%.

On surface the skips will dump into an elevated bin equipped with a truck loading chute. 55 t trucks will be loaded by the elevated bin and the material trucked to its destination on surface. Ore will be trucked to the processing plant and waste trucked to the WRSF, both some 2 km away from the shaft.

16.10 Mine Personnel

Mine development contractors will be utilized for mine construction and pre-production operations. The mine plan envisions, for budgetary purposes, three separate mine development contractors; one each for shaft sinking, underground development, and raise boring. Several existing open pit contract services will continue to support underground operations, including the batch plant and emulsion plant.

Development contractors will be replaced with an owner's team at the start of commercial production and take responsibility for all development and mining operations. Existing open pit employees will be trained and transitioned to the underground mine where possible.

All underground mine labour will operate on two 12-hour shifts, seven days per week. During mine construction contract labour will work a 14 day on, 7 day off work schedule. During mine operations underground labour will work 4 days on, 4 days off, equal to the current plant operators' schedule. Management, technical services, and contractor supervisory roles will work 5 days on, 2 days off where appropriate.

Total required mining labour is summarized in Table 16-15 and Figure 16-33. This includes all on-site and off-site crews.

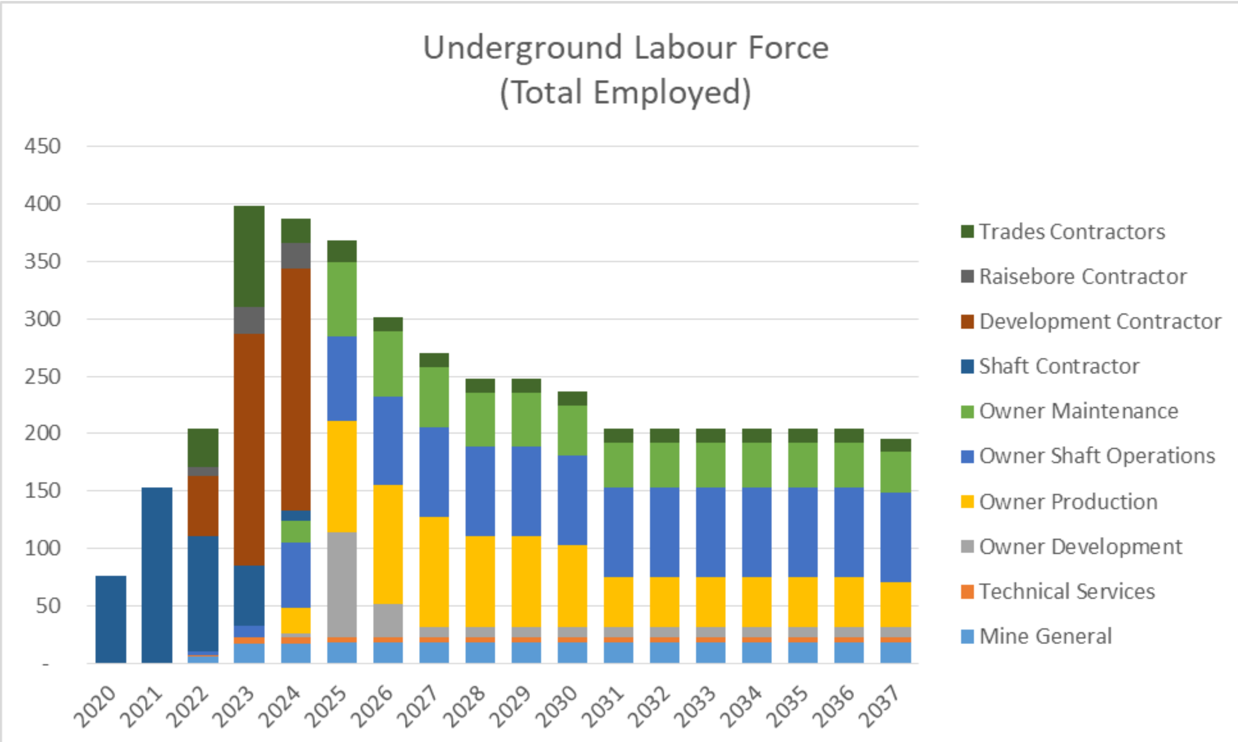
It should be noted that the current labour force carries all technical services and mine management under General and Administration costs, and this mine labour list only contains those positions in technical services and mine management that are required in addition to the current labour pool.

Table 16-15: Mine Labour Requirements

Manpower Compliment	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Owner																		
Mine General	0	0	6	17	17	18	18	18	18	18	18	18	18	18	18	18	18	18
Technical Services	0	0	2	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Development	0	0	0	0	3	91	29	8	8	8	8	8	8	8	8	8	8	8
Production	0	0	0	0	22	98	104	96	80	80	72	44	44	44	44	44	44	40
Material Handling	0	0	3	11	57	74	78	78	78	78	78	78	78	78	78	78	78	78
Maintenance	0	0	0	0	20	64	57	53	47	47	44	39	39	39	39	39	39	35
Total Owner Labour	0	0	10	33	124	349	290	258	236	236	225	192	192	192	192	192	192	184
Contractor																		
Shaft Sinking	76	154	101	52	9	0	0	0	0	0	0	0	0	0	0	0	0	0
Development	0	0	52	202	210	0	0	0	0	0	0	0	0	0	0	0	0	0
Raise bore	0	0	8	23	23	0	0	0	0	0	0	0	0	0	0	0	0	0
Trades	0	0	34	89	21	19	12	12	12	12	12	12	12	12	12	12	12	12
Total Contract Labour	76	154	194	366	263	19	12	12	12	12	12	12	12	12	12	12	12	12
Camp Space																		
Total Beds	56	113	131	226	211	193	159	144	133	133	127	111	111	111	111	111	111	107

Source: JDS (2019)

Figure 16-33: Underground Labour Force



Source: JDS (2019)

Average labour requirements are listed at three periods within the mine life; during pre-production, during drill and blast operations, and during the final draw down of the lobe.

16.11 Mine Equipment

The mobile equipment fleet for KDM is diesel-powered, trackless, and rubber tired. Mine development contractors will be utilized during pre-production and will be responsible for supplying all mobile equipment required for construction. KDM will take over mine development and operations at the start of commercial production and will purchase the required mobile mining fleet.

Underground equipment requirements are built up based on the productivities (operating-hours) required for mining activities occurring within a given time period. As such, equipment requirements fluctuate throughout the mine life. Major equipment productivities used to estimate equipment requirements are as follows:

- Jumbo drilling: 75 m/hour;
- Longhole drilling: 12 m/hour;
- Bolter: 6-7 bolts/hour; and

- Mucking: 240 t/hour.

Peak equipment requirements for both the mine development contractor and owner's team is summarized in Table 16-16.

Table 16-16: Mobile Equipment Requirements

Equipment	Contractor Supplied (pre-production)	Owner Purchased (production)
LHD (17t/7.0m^3)	2	2
LHD (21t/8m^3)	2	3
FEL (15t/5.4m^3)	1	1
Truck (20t/10.2m^3)	3	
Surface Truck (60t/35.8m^3)	2	4
Jumbo - 2 Boom	3	1
Longhole Drill - ITH	2	5
Bolter	3	2
Cable Bolter	2	1
Shotcrete Sprayer	2	1
Small Explosives Truck	1	1
Large Explosives Truck	1	2
Transmixer	2	1
Scissor Lift	2	1
Fuel/Lube Truck	1	1
Mechanics Truck	2	
Electrician Truck	1	1
Boom Truck	1	1
Grader	1	1
Telehandler	1	1
Personnel Carrier	1	1
Supervisor Truck	8	6
Utility Vehicle	10	6
Ambulance	1	1

Source: JDS (2019)

16.12 Mine Schedule

The shaft sinking schedule was provided by UMS. JDS completed the remainder of the development schedule along with the drilling and blasting schedule for the stope shapes. This was then provided to ITASCA to model the production draw schedule.

The project consists of a five-year pre-production period and a 13-year operating period.

The criteria used for scheduling the underground mine at KDM are as follows:

- The mine will operate two 12-hour shifts per day, 360 days per year;
- An average annual mill feed production rate of 2.6 Mt/a was scheduled, including ore from development and stopes; and
- Production ramp up over 6 months at 15% increments.

Shaft sinking commences in Q3 2020 and is completed by Q4 2022. Lateral development begins once the shaft sinking is complete. Production ramp up begins Q3 2024 with production commencing in Q1 2025.

16.12.1 Scheduling Philosophy

Shaft stinking commences as detailed engineering is completed and equipment is acquired. The ventilation shaft requires less engineering than the P/S shaft and is ready for pre sink by Q3 2020, followed by the P/S shaft shortly after. The 680 L dewatering gallery is the first priority in the schedule to ensure adequate time to dewater and depressurize the kimberlite pipe. The ventilation shaft has faster sinking rates and is able to reach the 680 L sooner, therefor, the dewatering gallery is established from the 680 vent shaft station.

Ventilation networks are established as the shafts are sunk at the 680 L, 480 L and 310 L. When lateral development commences on the 310 L, the first priority is establishing the ventilation network by the maintenance shop to ensure adequate fresh airflow. Once that has been achieved, the crusher room excavation is the next priority. Installation and commissioning of the crusher occurs immediately after the excavation is complete. Once this is complete, the equipping of the P/S shaft can commence.

Throughout mine development, establishing the ventilation network is priority to ensure fresh air is delivered to the working faces.

Three jumbos will be utilized during the pre-production period. All three jumbos will commence development on the 310 L. After majority of the development is complete, one jumbo will move to the 480 L and another to the 680 L. The last jumbo will remain on the 310 L for any rehabilitation work that needs to be completed throughout the mine life. The pre-production period lasts for 5 years including the 6 month ramp up.

16.12.2 Mine Development Schedule

Deswik scheduling software was used to optimize the mine development schedule. The shaft sinking schedule provided by UMS was transferred into Deswik to combine the shaft sinking schedule with the development schedule.

16.12.2.1 Lateral and Vertical Development Rates

The following scheduling constraints were used in Deswik for all lateral and vertical development:

- Maximum three development crews
    - 3.5 m/d on priority headings, plus 2.5 m/d on auxiliary headings, to a maximum of 11 m/d per active jumbo
- One raise boring crew
    - Maximum of 2.5 m/d
- One drop raising crew
    - Maximum of 3.5 m/d

- Daily development capacity
  - 40 m/d during development period
  - 19 m/d during shaft equipping period

Lateral development is not able to commence until the shaft sink is fully complete.

The stope and development cycle times and productivities used for mine development and production scheduling were estimated from the first principles.

16.12.2.2 Shaft Sinking Rates

The shaft sinking schedule for both the P/S shaft and the ventilation shaft was completed by UMS. The sinking rates are dependent on the rock type and can be seen in Table 16-17.

Table 16-17: Shaft Sinking Rates

Rock Type	Unit	P/S shaft	Vent Shaft
Pre-sink	m/day	1.21	1.54
Basalt Zone	m/day	2.14	2.49
Mea-Arkose	m/day	1.81	2.08

Source: JDS (2019)

16.12.2.3 Underground Infrastructure Installations

Underground infrastructure installations have been accounted for within the mine schedule. Table 16-18 outlines the installation time budgeted for each major piece of underground infrastructure. A combination of contractors, equipment vendors, and owner's team workforce will be utilized for infrastructure installations depending on the task and time period.

Table 16-18: Major Infrastructure Installation Durations

Infrastructure	Units	Duration (days)
Shaft Collar	ea	83
Shaft Station	ea	27 days if no development is required 54 days if development is required
Loading Pockets	lot	43
Drawbell construction	ea	5.25
Crusher and Conveyor Installation and Commissioning	lot	205
P/S shaft Equipping	lot	253
Pump Station	ea	20
Refuge Station	ea	18
Maintenance Shop	lot	40
Substation	ea	20

Source: JDS (2019)

16.12.2.4 Shaft Hoisting during Development

Both the production and ventilation shaft are used to hoist development muck at different points during mine construction. A 40 m/d lateral advance cap is placed on the development rates to account for the maximum hoisting capacity of the shafts when both production and ventilation shaft are available. During the equipping of the P/S shaft all development muck will be hoisted from the ventilation shaft and the development cap is reduced to 19 m/d during this period.

16.12.2.5 Mine Development Summary

Due to the mining method proposed at KDM the majority of the development needs to be complete before production can commence. All development related to the material handling system needs to be installed and commissioned before blasting of the drawbells can begin. Both shafts also need to be equipped and commissioned before ramp up can start. Total underground capital and sustaining development is 17.3 km and 1.85 km, respectively, equating to approximately 1.36 Mt.

There is approximately 16.3 km of lateral development and 2.8 km of vertical development underground. When all 3 jumbos are operational, an average of 750 lateral m/month is achieved. Mine development and milestones are summarized in Table 16-19 and Table 16-20, respectively.

Table 16-19: Mine Development Summary

Kimberlite Domain	Unit	Waste Development	Ore Development
Pre-production	m	10,499	6,771
Production	m	0	1,852
Total	m	10,499	8,623

Source: JDS (2019)

Table 16-20: Mine Development Milestone Summary

Milestone	Date
Shaft Sinking commences	Q3 2020
680 shaft station completed	Q3 2021
480 shaft station completed	Q1 2022
Shaft Sinking completed	Q4 2022
Lateral Development commences	Q3 2022
Ventilation network at maintenance shop established	Q4 2022
Crusher excavation commences	Q1 2023
Crusher installation and commissioning commences	Q2 2023
480 development commences	Q3 2023
680 development commences	Q3 2023
P/S shaft equipping commences	Q4 2023
Raise to surface from 680 complete	Q1 2024
Six month ramp up commences	Q3 2024

Source: JDS (2019)

16.12.3 Mine Production Schedule

16.12.3.1 Schedule Optimization

Itasca prepared a drawdown simulation using REBOP software to predict the blending effect of the different mineral zones and grades. The results of the REBOP simulation were used as a guidance to schedule underground production from the mine.

16.12.3.2 Production Rates

The following scheduling constraints were used in Deswik for all production activities:

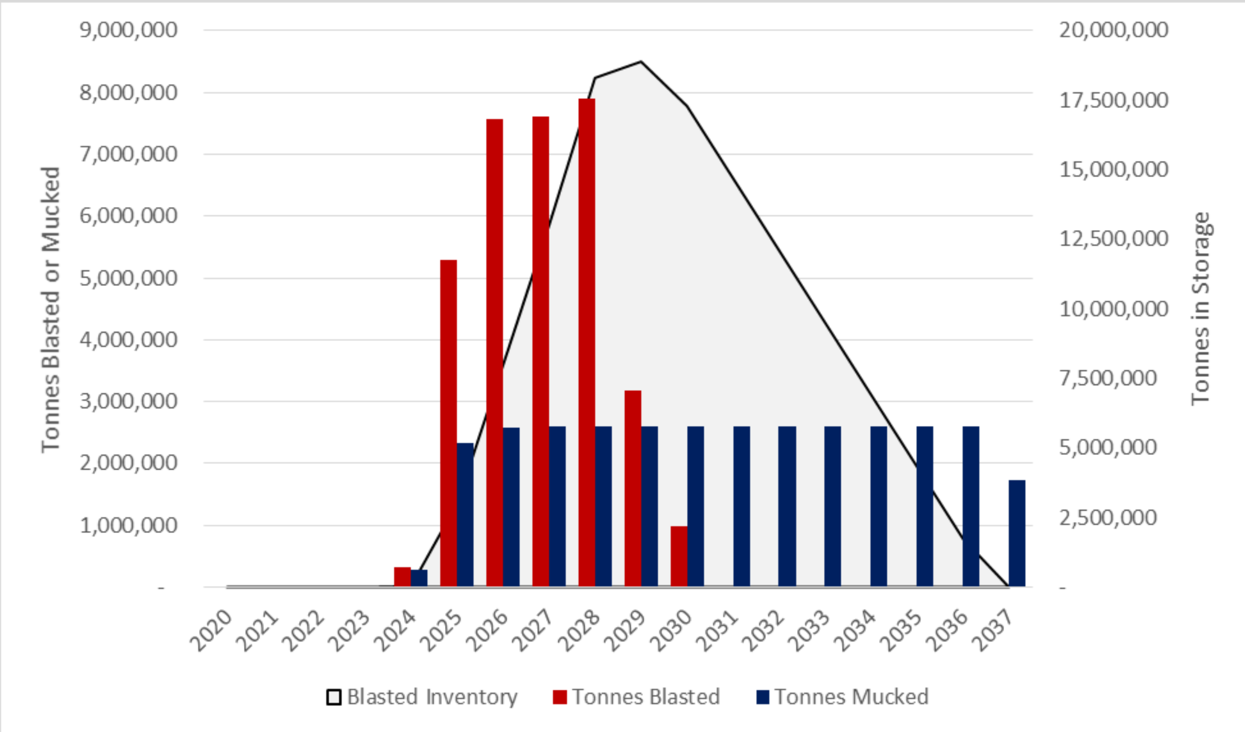
- Maximum of 105 m per shift per ITH drill;
- Maximum blasting rate of 21,000 t/d; and
- Maximum mucking rate of 216,000 t per month.

16.12.3.3 Mine Production Summary

Mine production of 7,200 t/d will be provided by draw down of the muck pile along with ore development during the production period.

Mine production commences in Q1 2025 after a 6 month ramp up in 15% increments. Five ITH drills will be utilized to drill and blast approximately 21,000 t/d in order to supply 7,200 t/d of swell to the draw bells for the first six years of operations. Peak broken inventory occurs in year five (2029) for a total of 18.9 Mt. After six years, the South Lobe will be fully blasted, and mucking will continue at a constant rate of 7,200 t/d until the underground reserves are depleted at the end of year 13 (2037). Figure 16-34 illustrates the relationship between the blasted inventory and mucked inventory over time.

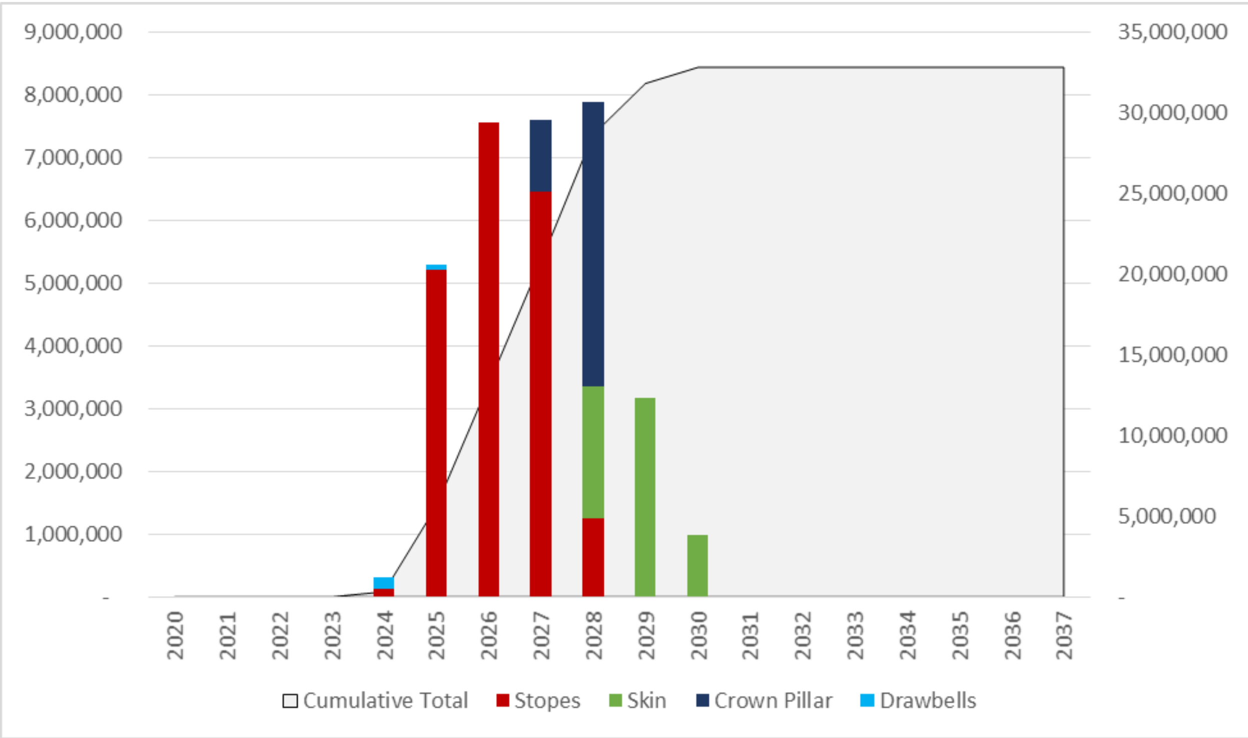
Figure 16-34: Blasting and Mucking Schedule Summary



Source: JDS (2019)

Drilling of the crown pillar commences Q1 2027 followed by drilling of the protective skin commencing in Q2 2027. Blasting of the protective skin does not commence until all of the crown pillar has been drilled and blasted in 2029. Figure 16-35 illustrates the blasting of the different production stope types over time.

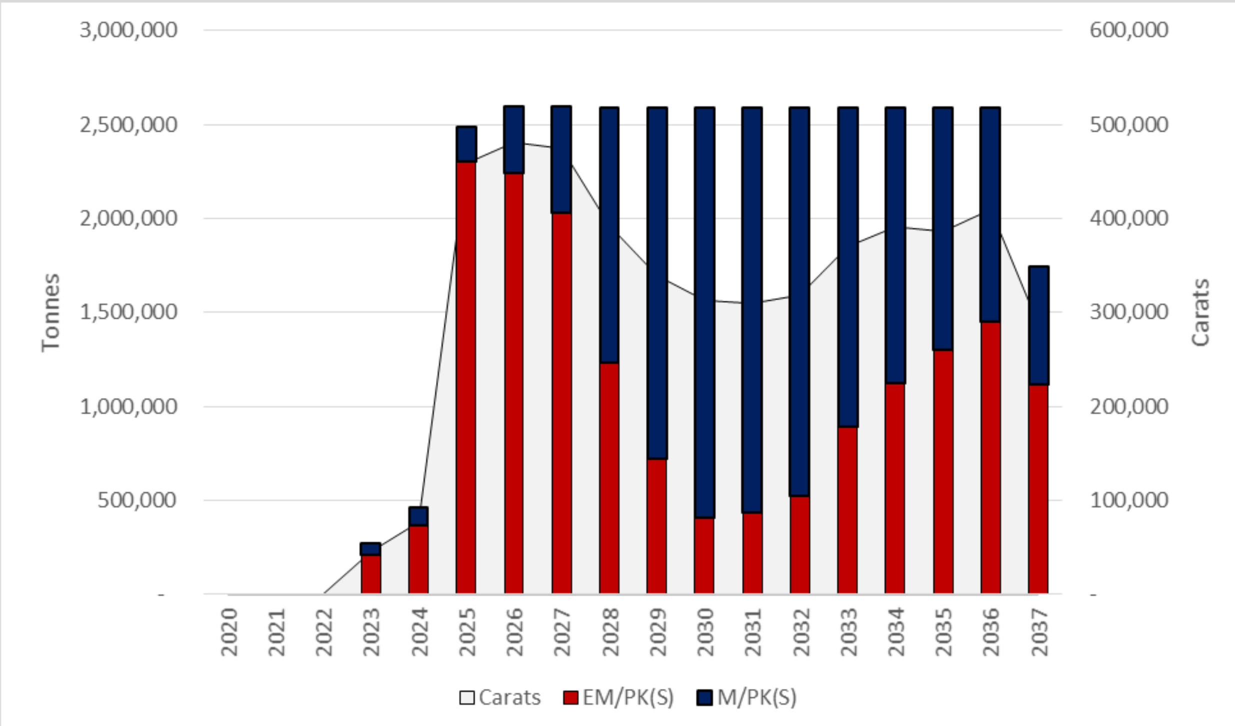
Figure 16-35: Blasting Schedule by Stope Type



Source: JDS (2019)

Figure 16-36 illustrates the breakdown between mineralized zones over time.

Figure 16-36: Hoisted Tonnes and Grade by Domain



Source: JDS (2019)

16.12.4 Underground Production Schedule

A number of schedule iterations and manual adjustments were made to produce a robust, sensible, and realistic schedule.

Final results of the Deswik schedule were organized such that physical metres, tonnes and carats were broken down into different categories for direct use in the cost model. From the final schedule, cost model requirements including items such as the mining fleet, workforce, consumables, ventilation, pumping, and power were determined and used to develop costs from first principals. Reports were generated monthly and then summed into annual totals for financial modeling.

The annual mine production schedule provided in Figure 16-37 shows annual summaries of ore and waste tonnage mined, ore grades and carats. Ore and waste tonnages have been rounded to the nearest million.

Table 16-21: Summary of Mining

Parameter	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Summary of Development																				
Shaft Development	km	1.5	0.1	1.1	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	0.0	0.0
Lateral Development	km	16.3	0.0	0.4	2.6	8.9	2.1	2.2	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Internal Raises	km	1.3	0.0	0.0	0.2	0.6	0.6	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
Metres / month / jumbo	m/m/j	185	0	0	223	258	140	180	49	0	0	0	0	0	0	0	0	0	0	0
Lateral Daily Advance	m/d	19	0	1	7	24	6	6	0	0	0	0	0	0	0	0	0	0	0	0
Summary of Drill & Blast																				
Development Ore	Mt	0.6	0.0	0.0	0.0	0.3	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
LH Shrinkage Stoping	Mt	20.9	0.0	0.0	0.0	0.0	0.3	5.3	7.6	6.5	1.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Crown Pillar & and Skin Wrecking	Mt	12.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.1	6.7	3.2	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Production Rate	kt/d	18.4	0.0	0.0	0.0	0.8	1.3	14.9	20.7	20.9	21.6	8.7	2.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Summary of Inventories																				
Drilled Inventory	Mt		0.0	0.0	0.0	0.1	0.0	2.9	2.3	6.7	1.2	0.4								
Blasted Inventory	Mt		0.0	0.0	0.0	0.0	0.0	3.0	8.0	13.0	18.3	18.9	17.3	14.7	12.1	9.5	6.9	4.3	1.7	
Summary of Production																				
Hoisted Ore	Mt	33.5	0.0	0.0	0.0	0.3	0.5	2.5	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	1.7
EM/PK(S)	Mt	16.3	0.0	0.0	0.0	0.2	0.4	2.3	2.2	2.0	1.2	0.7	0.4	0.4	0.5	0.9	1.1	1.3	1.4	1.1
M/PK(S)	Mt	17.1	0.0	0.0	0.0	0.1	0.1	0.2	0.4	0.6	1.4	1.9	2.2	2.2	2.1	1.7	1.5	1.3	1.1	0.6
Hoisted Grade	cpht	15.1	0.0	0.0	0.0	16.9	16.6	18.4	18.5	18.3	15.2	13.1	12.1	12.0	12.3	14.3	15.1	14.9	15.8	16.5
EM/PK(S)	cpht	19.9	0.0	0.0	0.0	18.4	18.2	19.1	19.8	20.4	20.2	19.8	19.7	20.3	20.9	21.3	20.6	19.1	19.7	19.7
M/PK(S)	cpht	10.6	0.0	0.0	0.0	11.1	10.3	10.4	10.6	10.8	10.6	10.5	10.7	10.3	10.1	10.6	10.9	10.7	10.8	10.8
Hoisted Carats	kc	5,053	0	0	0	45	77	459	481	475	393	339	313	310	318	370	391	386	410	287
EM/PK(S)	kc	3,246	0	0	0	39	67	440	443	413	249	142	80	88	109	190	232	248	286	219
M/PK(S)	kc	1,807	0	0	0	6	10	19	38	62	144	197	233	221	210	180	160	138	124	68

Source: JDS (2019)

16.12.5 Combined Open Pit & Underground Production Schedule

The open pit and underground mine production schedule for KDM incorporates the North and Centre lobe reserves mined from the open pit, and the South lobe reserves mined from both open pit and underground operations. The mill-feed tonnage will be provided from the open pit and existing stockpiles, until the underground reaches commercial production at the start of 2025. The open pit will operate until mid-2025; during the first half of 2025, mill-feed tonnage will be simultaneously provided from open pit and underground operations. Open pit and underground material will be stockpiled as needed when mine production exceeds mill capacity. Existing surface stockpiles will be consumed as processing capacity comes available.

The open pit mine production schedule corresponds to the Life of Mine (LOM) schedule and end of period maps prepared by Lucara in September 2019, using the previous mineral resource estimate. The LOM end of period maps were used to update the production schedule with the updated 2019 Mineral Resource. The open pit design and mining schedule was not optimized based on the updated 2019 Mineral Resource.

The mill blend and stockpiling strategy was based on the following criteria:

- Mill feed is prioritized based on value / tonne;
- UG feed is a mix of EM/PK(S) and M/PK(S) as underground material handling operations do not allow for selectivity between ore domains;
- UG ore is stockpiled until the start of commercial production in 2025; and
- Mixed stockpiles (contact & LOM) are processed at the end of the mine life.

Table 16-22 summarizes the combined LOM production schedule for KDM, including the open pit and underground mines, the mill feed schedule, and stockpile balances.

Table 16-22: Combined LOM Production Schedule

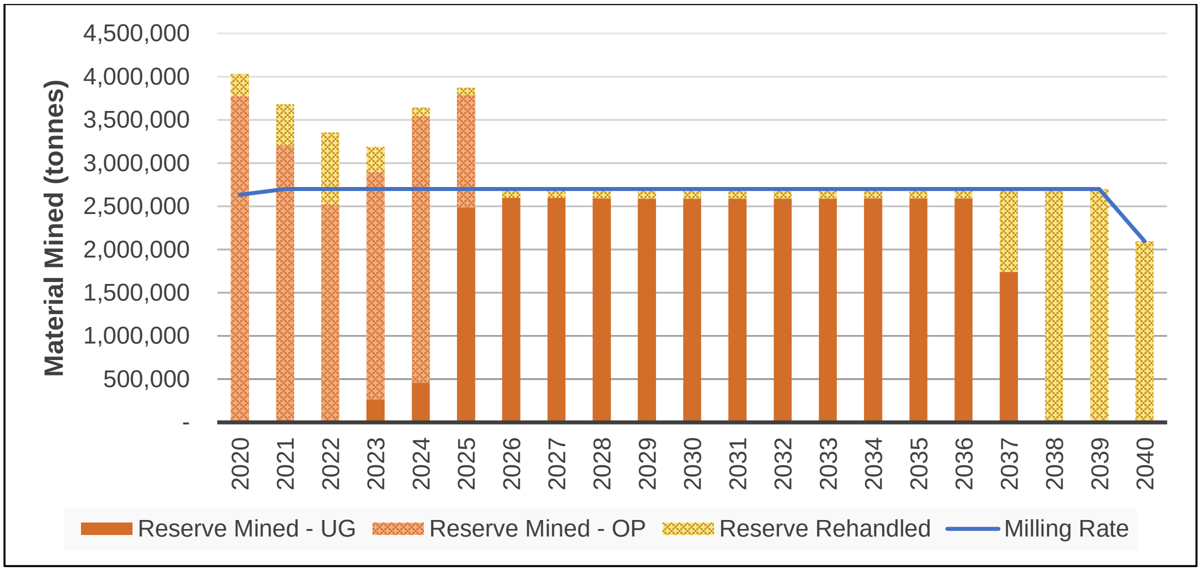
			Year Summary																				
Description	**Unit**	**Total**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**	**2040**
Mining Summary																							
Waste - OP Mining	Mt	12.7	4.0	2.4	2.4	2.1	1.6	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore - OP Mining	Mt	16.5	3.8	3.2	2.5	2.6	3.1	1.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore - UG Mining	Mt	33.5	-	-	-	0.3	0.5	2.5	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	1.7	-	-	-
Mill Feed																							
Direct Feed	Mt	44.3	2.4	2.2	1.9	2.4	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	1.7	-	-	-
From Stockpiles	Mt	11.7	0.3	0.5	0.8	0.3	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.0	2.7	2.7	2.1
Total Mill Feed	Mt	56.0	2.6	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.1
	cpht	14.0	15.8	14.1	12.7	15.1	15.5	19.7	18.2	18.2	15.3	13.4	12.4	12.2	12.5	14.5	15.2	15.0	15.8	14.9	10.0	6.8	4.3
	000's ct	7,838	416	381	344	408	420	532	493	492	413	361	334	330	339	391	411	405	426	401	269	184	89
Mill Feed - By Domain																							
North	Mt	1.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.7	0.4	-
	cpht	12.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14.8	9.6	-
	000's ct	149	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	109	40	-
Centre	Mt	3.5	0.4	0.5	0.2	0.5	0.4	0.0	-	-	-	-	-	-	-	-	-	-	-	0.3	0.6	0.6	-
	cpht	14.6	17.9	19.2	18.2	16.7	14.5	16.0	-	-	-	-	-	-	-	-	-	-	-	13.0	13.0	8.1	-
	000's ct	506	65	92	40	80	54		-	-	-	-	-	-	-	-	-	-	-	44	83	48	-
OP-South-EM/PK(S)	Mt	3.4	0.7	0.5	0.4	0.6	0.8	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	cpht	24.1	20.1	23.2	24.5	25.3	25.0	28.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	000's ct	810	145	106	92	150	205	113	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP-South-M/PK(S)	Mt	10.8	1.5	1.8	2.1	1.6	1.5	-	-	-	-	-	-	-	-	-	0.1	0.1	0.1	0.6	1.3	-	-
	cpht	10.5	13.3	10.4	10.1	10.9	10.6	-	-	-	-	-	-	-	-	-	17.4	17.4	15.1	11.3	5.9	-	-
	000's ct	1,134	206	183	212	178	160	-	-	-	-	-	-	-	-	-	12	19	16	71	78	-	-
Mixed	Mt	3.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.7	2.1
	cpht	4.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.7	4.3
	000's ct	185	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	96	89
UG-South-EM/PK(S).	Mt	16.3	-	-	-	-	-	2.1	2.3	2.1	1.3	0.8	0.5	0.5	0.6	1.0	1.2	1.3	1.4	1.1	-	-	-
	cpht	19.9	-	-	-	-	-	18.9	19.5	20.3	20.2	20.0	19.8	20.1	20.7	21.2	20.7	19.1	19.7	19.7	-	-	-
	000's ct	3,246	-	-	-	-	-	400	454	428	267	162	99	106	127	209	239	248	286	219	-	-	-
UG-South-M/PK(S)	Mt	17.1	-	-	-	-	-	0.2	0.4	0.6	1.4	1.9	2.2	2.2	2.1	1.7	1.5	1.3	1.1	0.6	-	-	-
	cpht	10.6	-	-	-	-	-	10.5	10.5	10.8	10.6	10.5	10.7	10.3	10.1	10.6	10.9	10.7	10.8	10.8	-	-	-
	000's ct	1,807	-	-	-	-	-	19	39	63	146	199	235	223	211	182	160	138	124	68	-	-	-
Stockpile Inventory																							
North																							
HG	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	0.4	0.4	0.4	0.4	0.4	-	-
MG	Mt	-	0.3	0.4	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.1	-
LG	Mt	-	-	0.1	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-
VLG	Mt	-	-	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	0.0	0.0	-
Centre		-																					
HG	Mt	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	0.2	0.6	0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.6	-	-
LG	Mt	-	0.3	0.3	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	-
VLG	Mt	-	-	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	-
OP - South-EM/PK(S)																							
HG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VLG	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP - South-M/PK(S)																							
HG	Mt	-	-	-	-	-	-	-	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	-	-	-	-
MG	Mt	-	-	-	-	-	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4	-	-	-
LG	Mt	-	1.1	1.5	1.1	0.2	-	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	-	-	-
VLG	Mt	-	-	0.1	0.5	0.8	0.9	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	-	-
UG																							
EM/PK(S) + M/PK(S)	Mt	-	-	-	-	-	0.3	0.7	0.9	0.8	0.7	0.6	0.5	0.4	0.3	0.2	0.0	-	-	-	-	-	-
Mixed																							
Contact	Mt	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	-
LOM	Mt	-	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	2.1

			Year Summary																				
Description	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Total Stockpile	Mt	-	6.1	7.2	7.7	7.5	7.7	8.6	9.7	9.5	9.4	9.3	9.2	9.1	9.0	8.9	8.8	8.7	8.6	8.5	7.5	4.8	2.1

Source: JDS (2019)

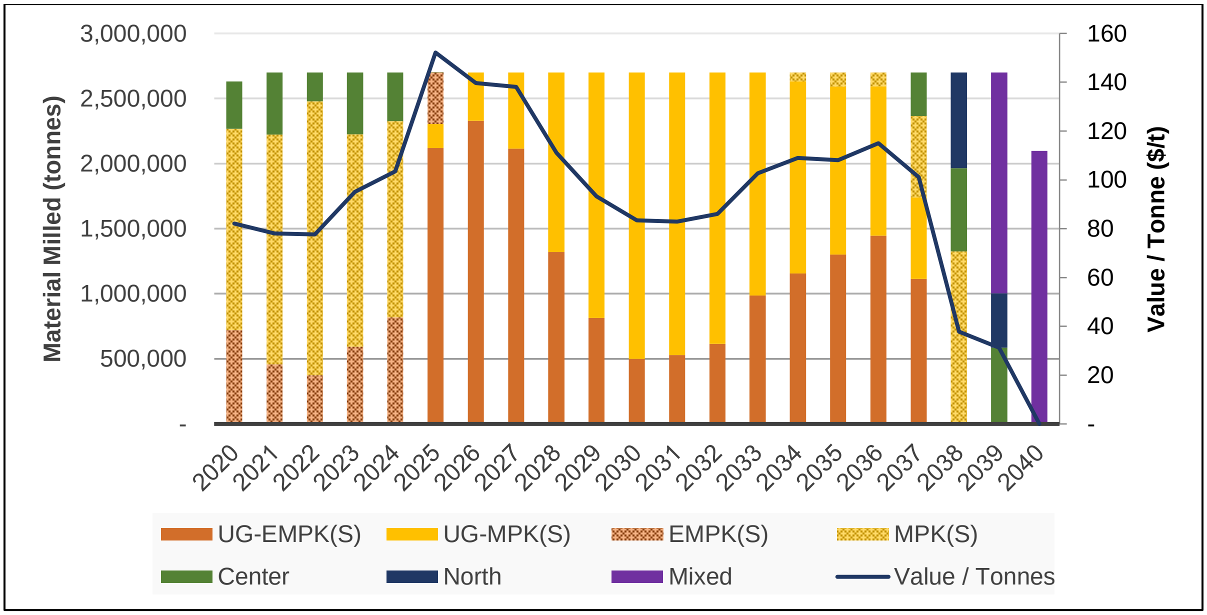
The total blended mine and mill feed from both underground, open pit, and stockpile operations is show in Figure 16-37 and Figure 16-38.

Figure 16-37: Summary of Mine Production



Source: JDS (2019)

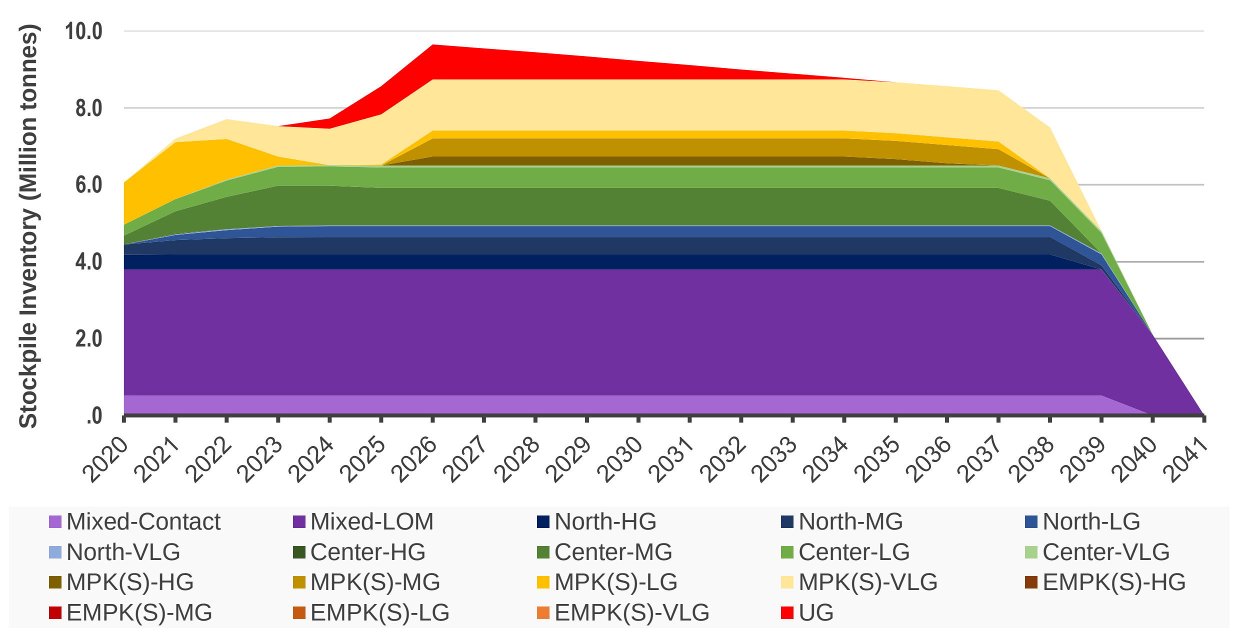
Figure 16-38: Summary of Mill Production



Source: JDS (2019)

A summary of the stockpile inventory opening balance is outlined in Figure 16-39.

Figure 16-39: Summary of Stockpile Inventory Opening Balance



Source: JDS (2019)

17 Process Description / Recovery Methods

17.1 Introduction

DRA Projects Pty Ltd. (DRA) was commissioned by JDS on behalf of Lucara to perform an overall treatment plant evaluation as part of a FS on extending the life of the Karowe Mine by mining underground after the completion of open pit (surface) mining.

To successfully assess current plant performance and production, a site visit was conducted on September 2 and 3, 2019 at KDM, Letlhakane, Central Botswana. 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 following sub-sections provide a brief historical summary associated with KDM since its inception in 2012.

17.1.1 Karowe Diamond Mine Phase I (Greenfields) History

Boteti Diamonds (a subsidiary of Lucara Diamond Corporation at that stage) contracted DRA Mineral Projects to provide complete Engineering, Procurement and Construction Management (EPCM) services for the design and construction of a diamond milling, Dense Media Separation (DMS) and recovery plant and associated crushing, screening and thickener systems for the Karowe Diamond Mine (called AK6 Mine at that time).

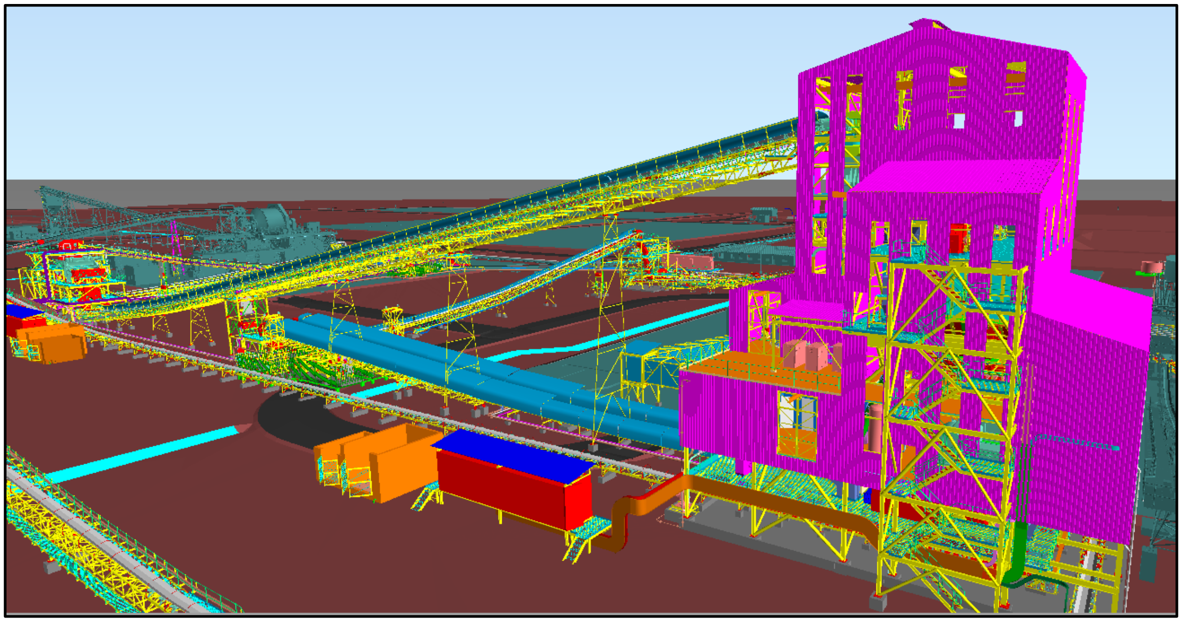
The Karowe Diamond Plant was designed to process 2.5 Mt of Run-of-Mine (ROM) kimberlite ore per annum with a single 200 t/h DMS module. The concentrate material from the DMS was subsequently treated through a 2.5 t/h wet X-ray Recovery for material reduction and diamond winning. Adequate space was allowed 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 the plant during Phase I was the autogenous 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 Karowe Diamond Mine Phase II (Brownfields) History

The brownfields Phase II Karowe Plant Upgrade Project was an expansion of the Phase I Greenfields AG Mill plant to cater for large diamond recovery up front in the circuit ahead of the DMS.

Figure 17-1: Model View of Karowe'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 about the KDM Phase II project was the utilisation of XRT machines in a large diamond recovery circuit to recognize and recover carbon-signature material (i.e. diamonds). By employing this technology in the process treatment plant, the top cut-off size of the plant could be significantly increased allowing for large stones to be recovered where previously they would have been broken in the pebble crusher and mill. In addition, XRT mitigated the impact of the high density of the Karowe kimberlite on the DMS performance as the DMS was limited to treating -8 mm material only.

What made the KDM Phase II project even more unique is the fact that XRT was also utilized in an audit function, where a portion of the -20 +8 mm tails from the main XRT building was treated through a single 50 t/h capacity downstream sorter for both metallurgical accounting and scavenging purposes.

Figure 17-2: Construction Completed and Fully Commissioned Karowe Phase II XRT Building

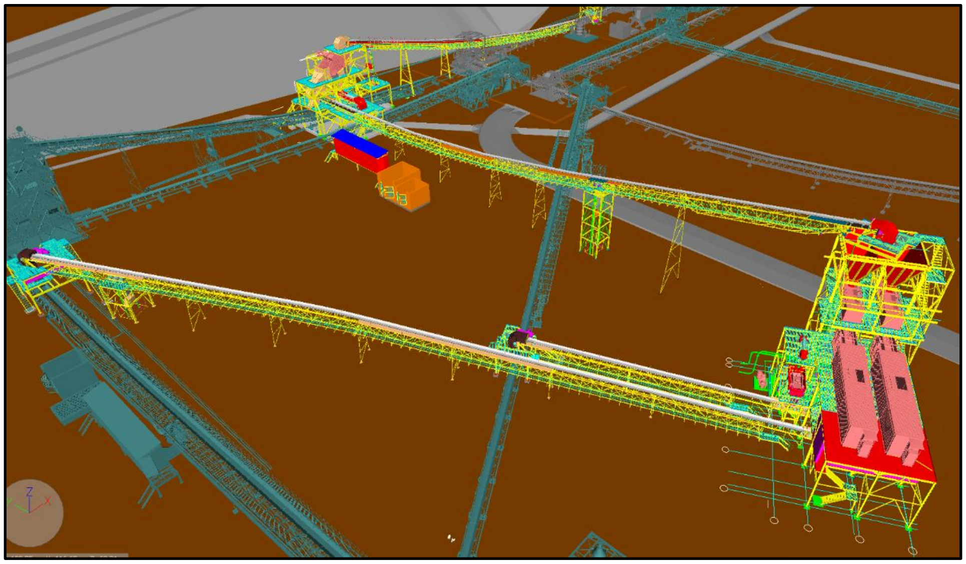
Source: DRA (2015)

17.1.3 Karowe Diamond Mine MDR 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. The aim was to sterilize the feed of liberated mega diamonds above 50 mm by adding a recovery step up front which was only top size limited by the available technology. A unique feature 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.

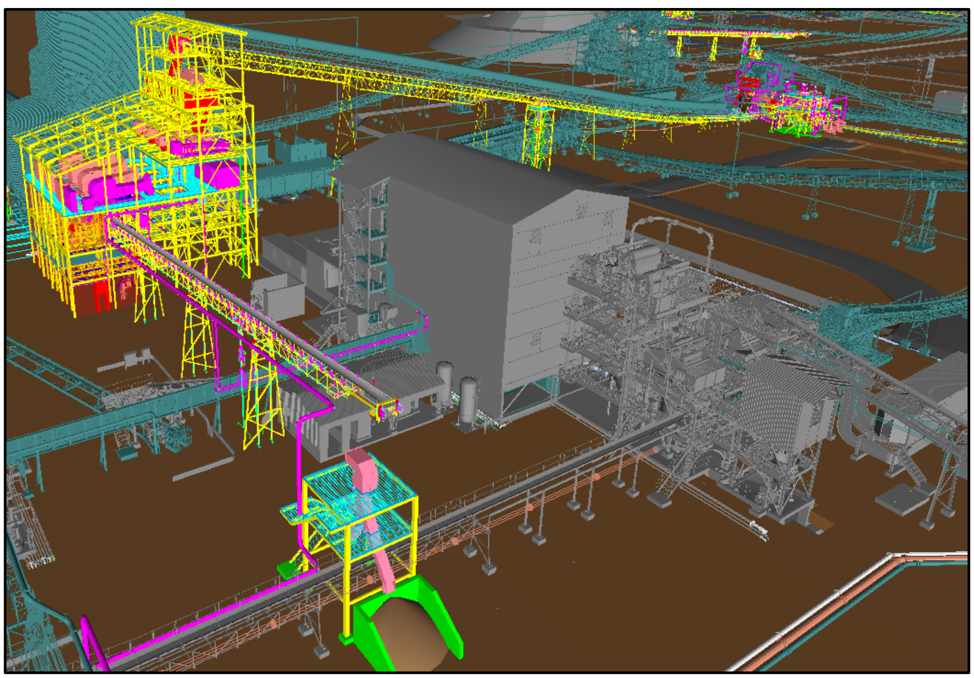
The Brownfields Phase III Karowe Plant Upgrade Project was another supplementary expansion to the KDM Phase II Expansion Project, providing complete EPCM services for the design, construction and commissioning of the Phase III brownfields expansion. Phase III made provision for the inclusion and application 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 Karowe's fresh, unweathered ore which could result in DMS yields in excess of ~25 %. A unique feature 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 was required due to the unique variance in ore body characteristics at Karowe, which has yielded some of the biggest diamonds in history – while at the same time having to negotiate one of the highest density and hardest kimberlites in existence.

Figure 17-3: Karowe MDR Project – 3D Model Snapshot



Source: DRA (2017)

Figure 17-4: Karowe Phase III Model Showing Primary XRT Machines



Source: DRA (2017)

17.2 Plant Design Criteria

The KDM Process Design Criteria (PDC) presented below is a high-level summary predominantly from the Phase I and II design and build.

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 test work 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 mtpa	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
Crushability Data				
Secondary Comminution				
Pre-crusher split	%	0 - 100	D3	A
Scalping screen cut size	mm	60.0	D3	A
Pre-crusher feed F100	mm	300	V	A
Crusher Type	-	Secondary Gyratory	D1	A
Closed side setting	mm	60 - 75mm	D3	A
AG Milling				
Discharge grate	-	TPL type grate	D2	A
Circuit Feed Size (Fresh Feed): F80	mm	~125.0	D2	A
Circuit Product Size: P80	mm	~37.5 - 50.0	D2	A
Circuit Product Size: % -1.5mm	%	~13 - 30	D1	A
Pinion Power (Mill Power)	kW	~3 045 - 3 783	D1	0

Criteria	Units	Value	Source	Revision
Installed Power	kW	4 000	D1	A
Mill Speed (Critical RPM)	RPM	14.6	D1	0
Mill Speed (% Nc)	% Nc	~80 - 82	D1	A
Circulating Load (% of Fresh Feed)	%	~5.5 - 12.5	D1	A
In Mill Density	% (v/v)	~68 - 70	D1	A
Product Slurry Density Target (-1.5mm, before dilution)	t/m^3	1.09	D1	A
Product Slurry Density Target (-1.5mm, before dilution)	% (w/w)	12.8	D1	A
Pebble Crusher and Bleed Screen				
Pebble Crusher Closed side setting	mm	25.0	D3	A
Bleed Screen Cut size	mm	32.0	D3	A
-32mm mill bypass	%	0, 12.5, 25, 37.5, 50, 62.5, 75, 100	D3	A
XRT Bulk Sorters				
Technology	-	XRT	D2	A
Size fraction: Middles	mm	8 - 14	D1	A
Size fraction: Coarse	mm	14 - 32	D1	A
Size fraction: Large	mm	32 - 60	D1	A
Diamond recovery (Large, Coarse % Middles)	%	≥ 98	C	A
Fines DMS				
Feed size	mm	1.5 - 8	D1	A
De-rated throughput	t/h	150 - 200	D1	A
Expected yields				
Average	%	7.40	D1	A
75th percentile	%	11.1	D1	A
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 +4mm	%	60.0	D2	A
-4 +1.5mm	%	40.0	D2	A
Average Yield Throughput				
+4mm	t/h	5.88	D1	A
+1.5mm	t/h	3.92	D1	A
75 Percentile Throughput				
+4mm - Middles	t/h	9.24	D1	A

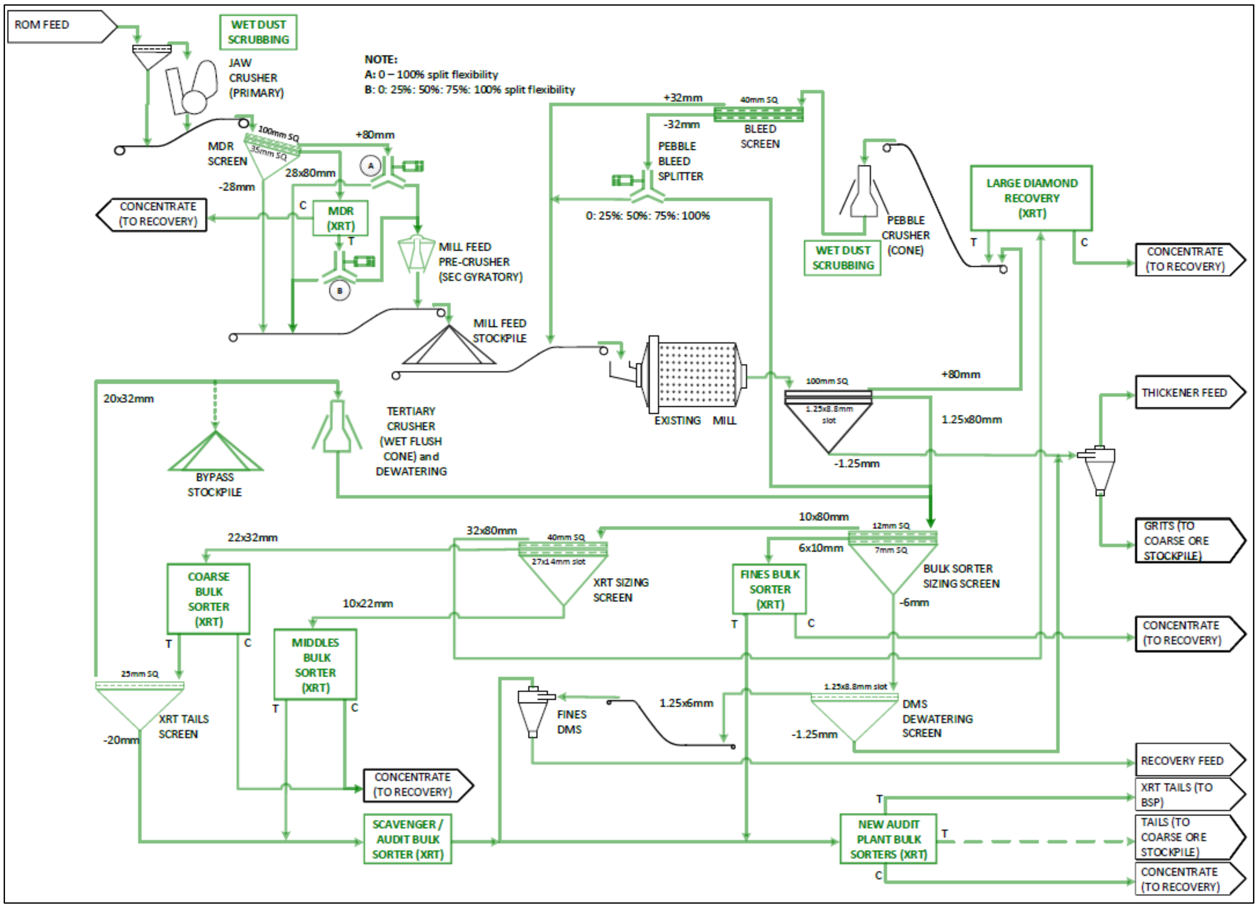
Criteria	Units	Value	Source	Revision
+1.5mm - Fines	t/h	6.16	D1	A
Wet MagRoll Capacity (Based on 2 Streams)				
+4mm - Middles 5 t/h	t/h	10.0	V	A
+1.5mm - Fines 3 t/h	t/h	6.00	V	A
MagRoll Reduction	%	65.0	D2	A
Wet X-Ray Capacity (Based on 2 Streams)				
+4mm - Middles 1950 kgh	t/h	4.00	V	A
+1.5mm - Fines 1050 kgh	t/h	2.00	V	A
Feed to X-Ray Circuit				
Average Yield +4mm - Middles	t/h	2.06	D1	A
Average Yield +1.5mm - Fines	t/h	1.37	D1	A
75 Percentile Yield +4mm - Middles	t/h	3.23	D1	A
75 Percentile Yield +1.5mm - Fines	t/h	2.16	D1	A
Reconcentration X-Ray Capacity				
+4mm - Middles	kgh	25.0	V	A
+1.5mm - Fines	kgh	10.0	V	A

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 most recent 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 kit from Phase I, while the equipment and streams highlighted in green font denote subsequent changes post-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 M/PK(S) and EM/PK(S).

Figure 17-5: Overall Karowe Diamond Mine Block Flow Diagram (Current)



Source: DRA (2019), updated following site visit September 2-3, 2019

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. The tabulated summary list includes all key equipment duties with installed drives noticeably equal to or larger than 100 kW; spanning from first treatment plant construction and commissioning in 2012 and covers all three phases of Greenfields first-built and Brownfields expansion projects.

Table 17-3: List of Major Components – Summary Mechanical Equipment List

Tag Number	Description	Specification	Installed Power (kW)
100-CJA-045	Primary Jaw Crusher	Size: CJ613	160
120-FCV-005	In Plant Stockpile Feed Conveyor	Width: 1200 mm	220
200-AGM-010	AG Mill	Size: 8.53 m Ø diam x 4 m long	4 000
200-PCB-030	AG Mill - Effluent Pump	Size: 10/8F-AH-5VCM	160
220-CCA-020	Pebble Crusher	Size: XL 400 Excel-Raptor (cone crusher)	300
300-PCB-045	Cyclone Feed Pump	Size: 10/8F-AH-5VCM	250
300-PCB-120	CM Pump	Size: 10/8F-AH-5VCM	160
500-PCB-090	Slimes Disposal Pump No. 1	Size: 8/6F-AH-6VCM (High Efficiency)	132
500-PCB-095	Slimes Disposal Pump No. 2	Size: 8/6F-AH-6VCM (High Efficiency)	132
500-PCB-100	Slimes Disposal Pump No. 3	Size: 8/6F-AH-6VCM (High Efficiency)	160
520-PCC-025	Mill Process Water Supply Pump	Size: NF200-500-P55	185
115-GGA-035A	Secondary Crusher	Model: KG4513 (Secondary Gyratory)	185
380-CCA-030A	Tertiary Crusher	Model: Cybas-i 1200 (wet flush cone crusher)	220
520-PCC-200	DMS/Bulk Sorter Process Water Pump	Size: NF200-400-P55	110

Source: DRA (2015)

Since the conclusion of all MDR and Phase III work at KDM (expansion phases concluded in 2017), the following main plant upgrades and initiatives have been noted during the recent September 2 and 3, 2019 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 2018 with the following noticeable observations made:
    - A vibrator (mechanism) was 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

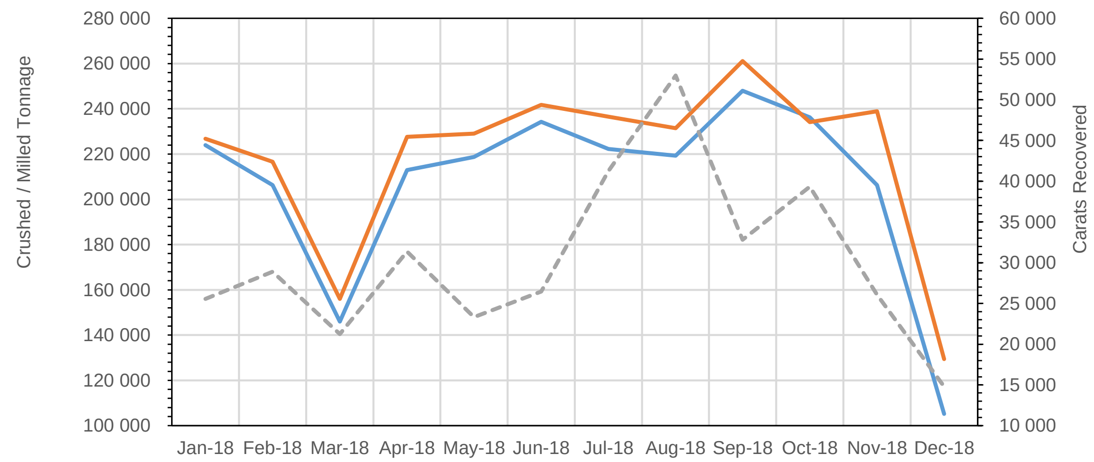
- An operational bypass flexibility option exists regarding two vibrating feeders post-new secondary gyratory crushing feed bin arrangement: the secondary gyratory crusher can be bypassed when associated downtime is experienced, or in case of excessive fines fed through the system (not purposely directed 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 anticipated for first half of next year (2020). Due to the prevalent nature of water in and around these machines (dribblings, spray water), this has led to subsequent corrosion of the units to the point where it has to be replaced/refurbished;
- Phase II audit XRT machine now utilised and incorporated as part of the mainstream plant in a primary "scavenger" application / duty;
- New XRT audit plant (at the back of the recovery plant) treating DMS, grits and XRT tails material was observed. An overflow (feed) chute arrangement draws down material and feeds the new XRT audit plant when the new XRT audit plant feed conveyor starts. Equipment noticed during the September 2-3, 2019 site visit walkabout includes:
  - Screen;
  - Bin;
  - Two XRT machines;
  - One collective Audit XRT Tails conveyor; and
  - Minus 4mm material is not treated through the new audit XRT plant but transported to the Bulk Sorter Plant (BSP) for subsequent treatment/processing.
- DMS/XRT floats (i.e. coarse residue stockpile) initiative: material from the coarse residue stockpile is earmarked for treatment through the BSP after finalization of an adequately defined drilling program as part of the Tailings (coarse residue) Resource Evaluation Program (TREP).
- Dust suppression system re-starting initiative: the existing dust suppression system has been restarted at the end of August 2019 using R/O plant filtered water quality to combat ore transfer point dust emissions;
- Current R/O plant capacity was expanded during November 2018 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);
- New raw and process water tanks, complete with new pump manifolds and pumps were installed and successfully commissioned in August 2017 as part of the Phase III implementation;
- Recovery magnetic roll (or MagRoll) separators were effectively bypassed on February 12, 2018 and consequently de-commissioned on September 5, 2019;
- XRT sorthouse upgrade was completed on December 3, 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. The following two graphs (Figure 17-6 and Figure 17-7) summarize 2018 plant performance in terms of crushed / milled tonnage, carats recovered and key treatment plant feed stream Particle Size Distribution (PSD) data.

Figure 17-6 shows that lower monthly production was observed for both March and December 2018. The decreased production in March 2018 was attributed to the reduced number of production days as a result of a five-day plant maintenance shutdown period. Similarly, in December 2018, the decreased production was as a result of a ten-day plant maintenance shutdown event incorporated during that particular period.

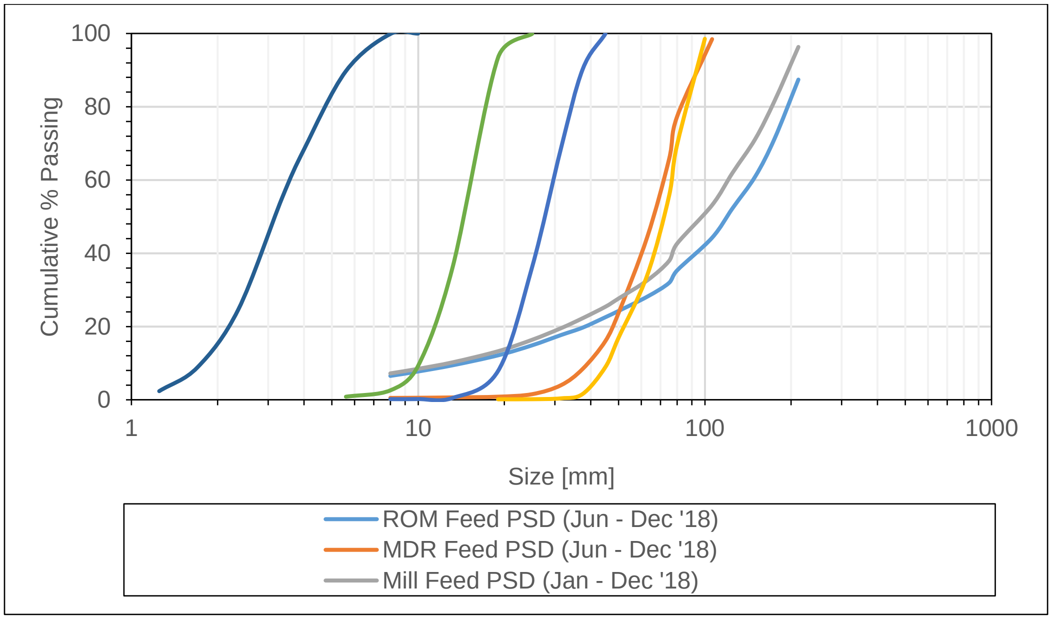
Figure 17-6: 2018 Crushed / Milled Tonnage vs. Carat Recovery



	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18
Crushed Tonnage	224 020	206 259	146 036	212 875	218 680	234 321	222 234	219 348	247 981	236 305	206 264	105 179
Milled Tonnage	226 842	216 547	156 018	227 606	229 003	241 694	236 468	231 386	261 108	234 074	238 901	129 401
Carats Recovered	25 563	28 881	21 240	31 378	23 301	26 469	41 256	52 976	32 799	39 347	26 063	14 790

Source: Lucara Botswana (2019)

Figure 17-7: 2018 Treatment Plant Key Feed Stream PSDs



Source: Lucara Botswana (2019)

The ensuing two tables (Table 17-4 and Table 17-5) present existing treatment plant panel aperture and crusher closed side (CSS) parameters.

Table 17-4: Key Screen Panel Aperture Summary

Screen Description	Screen Panel Aperture Size
MDR Screen (Double Deck)	Top: 100 mm SQ Bottom: 35 mm SQ
Bleed Screen (Single Deck)	40 mm SQ
Mill Discharge Screen (Double Deck)	Top: 100 mm SQ Bottom: 1.25 x 8.8 mm SLOT
Bulk Sorter Sizing Screen (Double Deck)	Top: 12 mm SQ Bottom: 7 mm SQ
XRT Sizing Screen (Double Deck)	Top: 40 mm SQ Bottom: 27 x 14 mm SLOT
XRT Tails Screen (Single Deck)	25 mm SQ
DMS Dewatering Screen (Single Deck)	1.25 x 8.8 mm SLOT

Note: "SQ" denotes square aperture and "SLOT" denotes slotted aperture
Source: Lucara Botswana (2019).

Table 17-5: Crusher CSS Summary

Crusher Description	Closed Side Setting (CSS) Size (mm)
Primary Jaw Crusher	180
Secondary Gyratory Crusher	65 – 90
Pebble Crusher	35 – 38
Tertiary Wet Flush Crusher	14

Source: Lucara Botswana (2019)

Another aspect that has been identified (apart from the desktop study completed in 2018) when considering underground mining and operations will be that of water management and potential impact(s) on the overall macro water balance when encountering water at depth.

17.4 Process Plant Description

17.4.1 Crushing

Previous mill simulations and associated mass balances indicated that to achieve a head feed rate of 350-500 t/h processing hard ore, a secondary crushing stage is required ahead of the mill. The secondary crushing section stabilizes and reduces the mill load as well as the pebble crusher load. It also assists with top size feed control to the downstream milling section.

ROM material is delivered to the ROM tip by means of articulate and non-articulated trucks and first stage crushing in the form of a primary jaw crusher reduces ore to an acceptable feed envelope size ahead of the secondary crusher section.

Depending on the material treated, a proportion or the entire primary crushed ROM stream is diverted and processed through the secondary crusher circuit. Feed to the secondary crusher is scalped of undersize on the MDR screen while the oversize removed on the same screen is partially sent to the crusher depending on a diverter setting. In addition, a portion (or all or none) of the MDR tails can be sent to the secondary crusher. The secondary crusher product is reintroduced onto the mill stockpile feed conveyor with the screen undersize and bypass stream.

The +80 mm mill screen product and the 32 x 80 mm LDR XRT tailings are processed through the existing pebble crusher. The pebble crusher product is sized at 32 mm with all the +32 mm material reporting to the mill feed conveyor. A portion of the -32 mm material bypasses the mill with the split balance of the -32 mm bleed screen undersize reporting directly to the mill feed conveyor. The bleed is required and balanced operationally to reduce mill loading.

The 20 x 32 mm tailings from the XRT bulk sorters are processed through a wet flush tertiary crusher circuit to liberate diamonds in this particular size fraction. The tertiary crusher product is reintroduced back into the circuit via a bulk sorter sizing screen and reports to the relevant downstream process based on the crushed product size envelope.

17.4.2 Comminution – Milling, Bleed Screening & Pebble Crushing

Fresh mill feed is introduced into the mill from the feed stockpile along with a variable portion of the pebble crusher product directly. A bleed screen has been installed on the pebble crusher product stream, so that a portion of the – 32 mm pebble crusher product can be bled out of the mill feed and report directly to downstream processes, thereby alleviating and balancing mill loading. The current AG Mill discharge grate incorporates Turbo Pulp Lifter technology to improve discharge and grate efficiency as well as withdrawal of material out of the mill.

17.4.3 XRT

The mill screen product (1.25 mm x 80 mm) is sized on the bulk sorter sizing screen and XRT sizing screen with the 32 mm x 80 mm oversize size fraction reporting to the LDR XRT section. The purpose of the LDR is to recover large diamonds before the stream is processed through the pebble crusher circuit. The 22 mm x 32 mm and 10 mm x 22 mm size fractions report to the coarse and middles bulk sorter sections respectively. The LDR XRT tailings are processed through the pebble crusher circuit. XRT tailings from the coarse bulk sorters are transported to the tertiary crusher – passing over the XRT tailings screen first to separate the -20 mm size fraction. The combined coarse and middles bulk sorter tailings report to the scavenger audit XRT and then have the option to be either diverted to the new XRT audit plant or to be discarded as final coarse tailings on the DMS floats coarse ore stockpile.

17.4.4 DMS

As hard, high specific gravity (SG) material is encountered from an ore treatment perspective, the denser the material becomes. High yields result in higher DMS cyclone sink throughputs to the recovery circuit which create a bottleneck for the recovery plant. The existing fines DMS plant processes the 4 mm x 1.25 mm size fraction and beneficiates diamondiferous concentrate from less heavy reject / gangue material. The fines DMS throughput has been de-rated to accommodate the shift in current feed size treatment.

17.4.5 Recovery

The existing recovery plant processes the 5-6 mm x 1.25 mm size fraction received from the DMS section. In order to accommodate intermittent hard, high DMS-yielding ore types (MP/K(S), EMP/K(S), a bulk reduction stage using MagRolls was initially added and incorporated as part of the original design. Since September 5, 2019 however, the MagRolls have been de-commissioned due to the conversion of the DMS plant from coarse to fines treatment (i.e. seeing less throughput) and due to the very low prevalence of magnetic diamond-bearing material observed in the DMS sinks yield portion ultimately reporting to the recovery plant. Other noticeable equipment located inside the recovery plant consists of wet x-ray machines, Infra-red (IR) drier and a dry reconcentration X-ray luminescence machine.

17.4.6 DMS Residue and Effluent Disposal

DMS residue, XRT residue and degrit screen grits are discarded as final coarse residue on the DMS floats coarse residue stockpile. Alternatively, a split of this material can be fed to the new XRT audit plant.

All effluent streams generated in the plant (-1.25 mm) are pumped to the degrit effluent cyclones situated at the thickener. Overflow from the cyclones gravitates to the thickener feed well where flocculant at the correct solution strength is introduced to agglomerate and consolidate ultrafines for final disposal / removal to the FRD via the fine residue disposal pump train. Underflow from the cyclones reports to the degrit screen for fines dewatering and disposal to the DMS floats coarse ore stockpile.

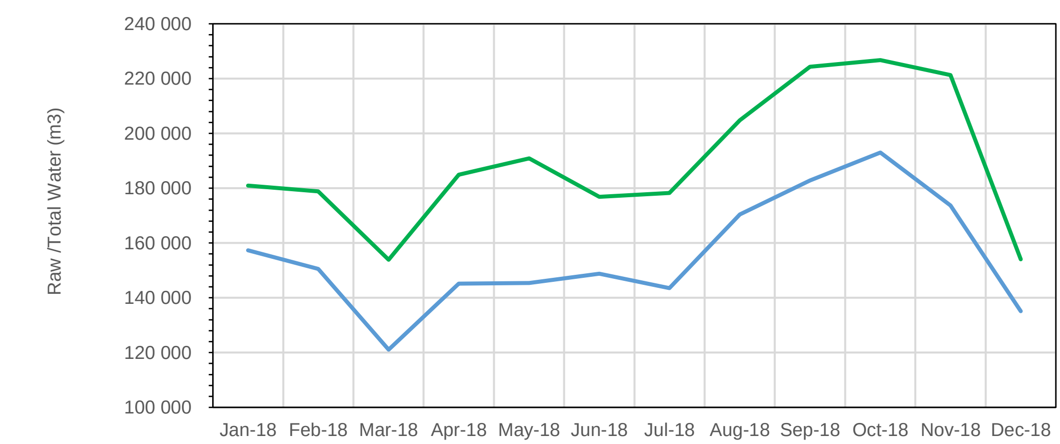
17.4.7 Services

Current plant-wide services at KDM's process treatment plant include instrument and process air from the respective compressors for valve actuation and XRT air-blow. Process water is collected and recycled back into the plant via thickener and process water tanks. Raw water is supplied to various end-users requiring borehole quality water for conversion to R/O potable or filtered water quality via the existing (and newly expanded) R/O plant for specific duties. Water chillers in the XRT and recovery sections continuously cool down equipment. Dust suppression will be recommissioned 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 2018 is graphically presented in Figure 17-8. Raw water to the process treatment plant is supplied from pit dewatering and wellfields sources. In early 2019, the wellfiled boreholes were discontinued, all water is now sourced from pit de-watering boreholes.

Figure 17-8: 2018 Karowe Raw / Total Water Consumption



	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18
Raw Water (m3)	157 272	150 536	121 076	145 143	145 391	148 841	143 525	170 409	182 874	193 013	173 683	135 136
Total Water (m3)	180 890	178 841	153 887	184 918	190 854	176 874	178 272	204 809	224 357	226 746	221 237	154 061

Source: Lucara Botswana (2019)

17.4.9 Water Management

17.4.9.1 Objectives

An integrated mine water balance model was developed that could be used as a continuous simulation to:

- Quantify the current and future mine water surplus and deficit conditions;
- Quantify the surplus water and evaluate the adequacy of the supply line quantity and quality constraints;
- Evaluate the impact of storm water conditions during 1:50 and 1:100 wet conditions to determine the adequacy of contact water containment capacity and risk to the underground mine; and

- Quantify the build-up of mass in the process water and water quality deterioration with potential long-term impacts from mine residue facilities.

17.4.9.2 Results

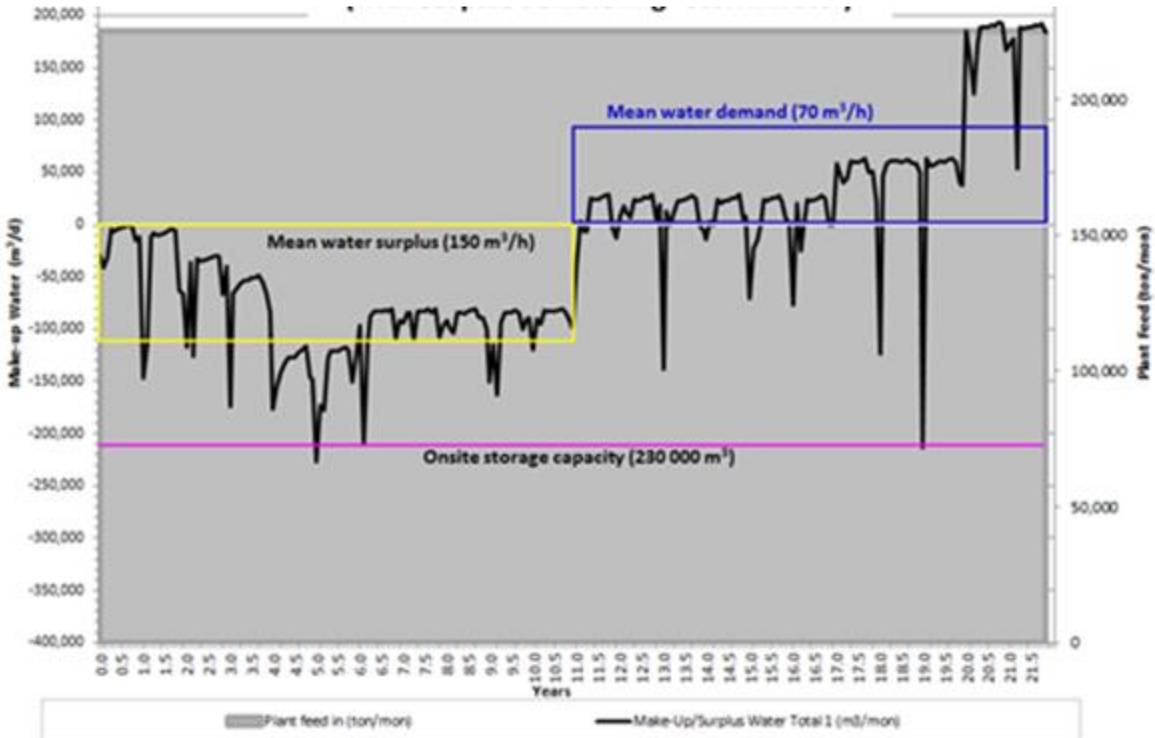
The mine make-up water requirement (demand) ranges between 195 m^3/h to 250 m^3/h (Figure 17-9). Mine dewatering at 350-400 m^3/h produces surplus water of 125 m^3/h to 175 m^3/h which is discharged to a local water consumer (Figure 17-10).

An agreement is in place between Lucara Botswana and a local water consumer on the volume of water with a water quality constraint of 4,000 mg/L Total Dissolved Solids (TDS). After 8-12 years, the dewatering volumes and eventually decreases again to 150-200 m^3/h, which will cause a deficit. At that time, the backup wellfield will be re-established and tested to supplement future anticipated water supply demands.

The water ingress risks to the underground from 2030 (when it breaks through to the open pit) was simulated based on a 1:100 year wet event (180 mm in 4 days) that produces 80,000 m^3 (Figure 17-11). To mitigate this, provision was made for:

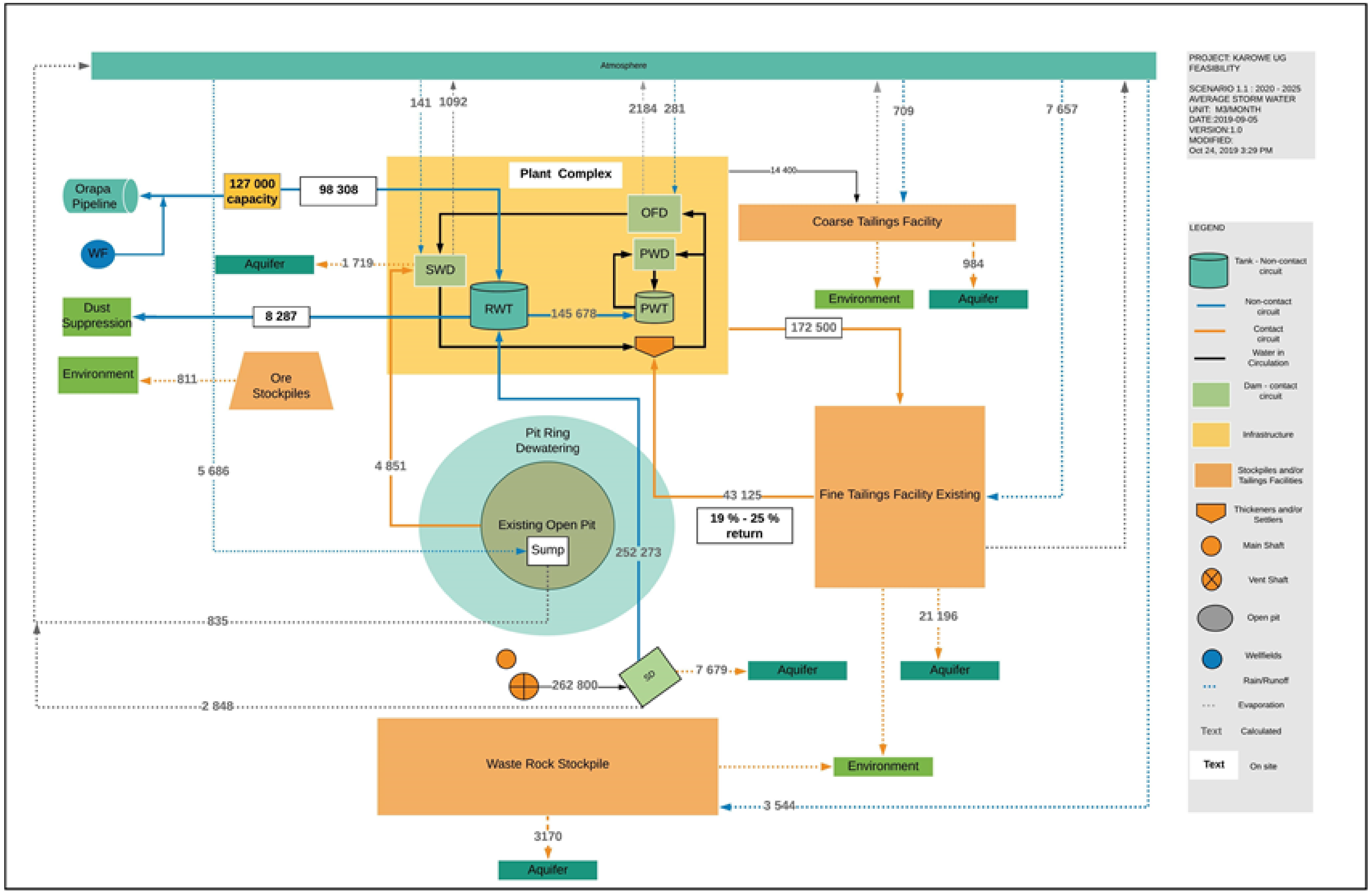
- On-ramp paddocks to retain 40 000 m^3;
- Underground storage in tunnels below 310 L of 35 000 m^3;
- Pumping capacity that shifts from 680 L to 310 L from 2030 at 800 m^3/h; and
- A surface settling dam of 40,000 m^3.

Figure 17-9: Scenarios 1 & 2 LOM Make-up Water Demand Curve (2020 – 2040)



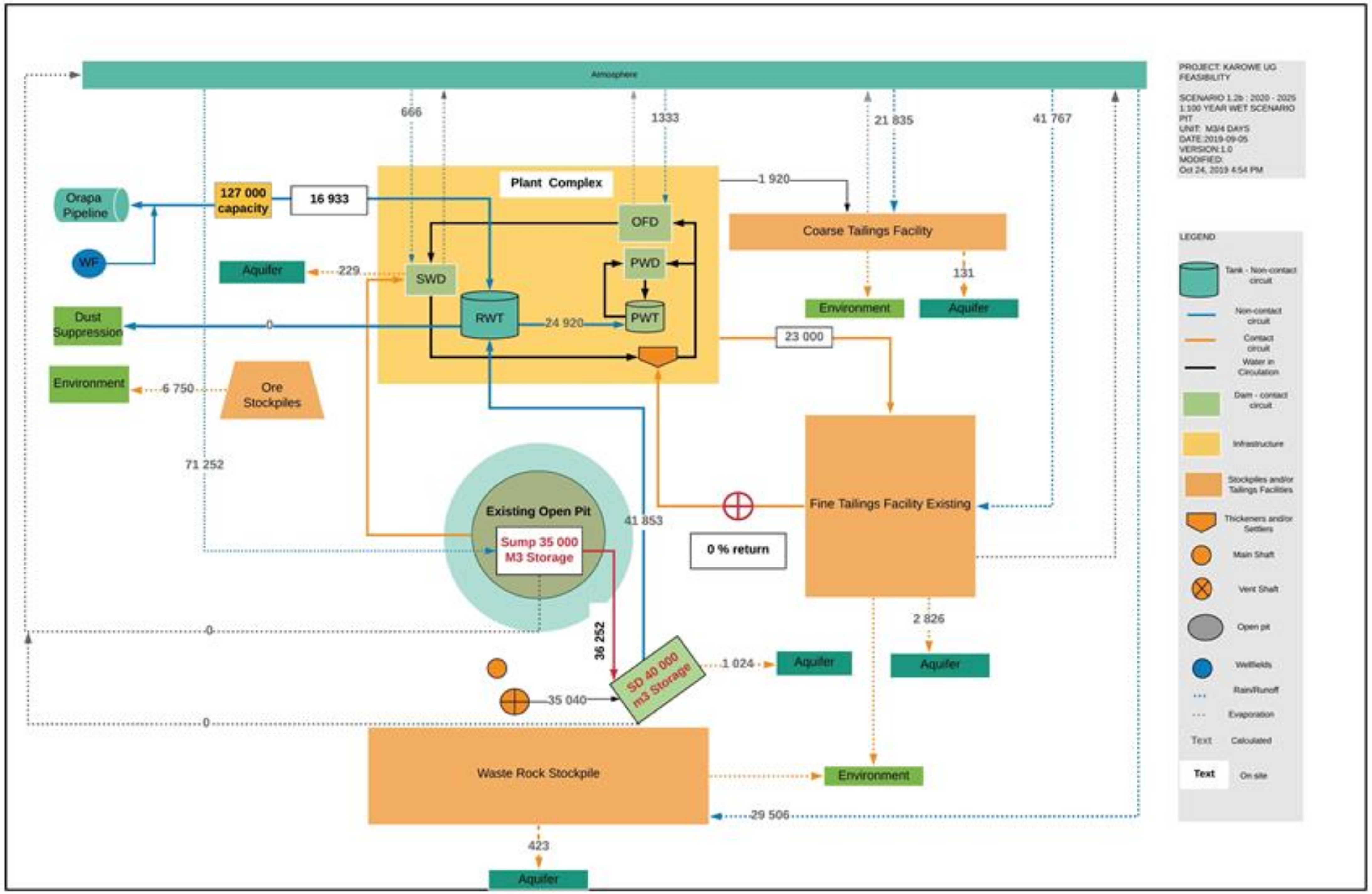
Source: Exigo (2019)

Figure 17-10: Mine Water Balance: Scenario 1.1: Average Monthly Flows 2020 – 2025 OP & UG



Source: Exigo (2019)

Figure 17-11: Mine Water Balance: Scenario 1.2b Flows 2020 – 2025 OP and UG @ 1:100 wet



Source: Exigo (2019)

17.4.9.3 Recommendations

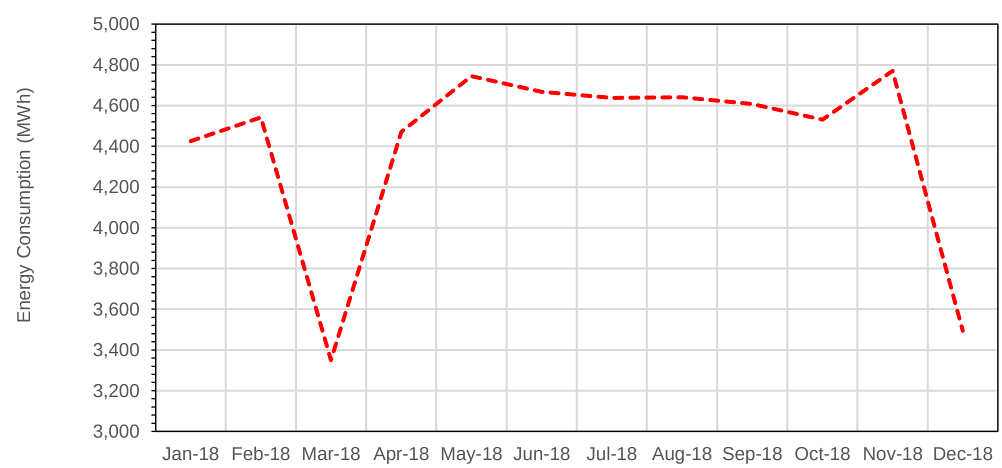
The following are recommendations with respect to water management as part of the detailed engineering program in the Recommendation Section:

- Value engineering with the mine water and mass balance model to optimize plant-fine residue circuit water consumption required underground and surface storages capacities for flood conditions and pumping rates;
- The supply line capacity should be increased to 250-300 m^3/h to allow for potential higher inflow rates from the underground galleries and fan drains if / when the mine zone aquifer conditions changes from confined to semi- or unconfined;
- A mine water and salt management plan must be developed to ensure sustainable water quantity and quality management; and
- The online water information management system (WIMS) should be integrated with the Supervisory Control and Data Acquisition (SCADA) for real time dewatering status and integrated mine water balance management.

17.4.10 Energy Consumption

Energy consumption data (associated with the process treatment plant) observed for the 2018 period is reported and summarized in Figure 17-12.

Figure 17-12: 2018 Karowe Energy Consumption



	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18
Energy (MWh)	4,426	4,541	3,350	4,471	4,745	4,667	4,637	4,641	4,607	4,531	4,770	3,493

Source: Lucara Botswana Internal Data (2019)

Above figure indicates lower energy consumption information for both March and December 2018 due to lower monthly production that was observed for those associated months. The decreased production in March 2018 was attributed to the reduced number of production days as a result of a five-day plant maintenance shutdown period. Similarly, in December 2018, the decreased production was as a result of a ten-day plant maintenance shutdown event incorporated during that particular period. Average power consumption in 2018 ranged between 4400 and 4800 MWh (excluding March and December 2018).

18 Project Infrastructure and Services

The UG Project will include the use of existing and new infrastructure at the Karowe Mine. Project infrastructure is designed to support the operation of a 2.6 Mt/a mine and 2.7 Mt/a processing plant. The UG Project will make use of existing infrastructure including the processing plant, site access road, airstrip, dewatering pipeline, maintenance facility and bulk fuel storage.

Existing infrastructure to be expanded or upgraded includes the potable water plant, sewage treatment facility, site substation and power distribution, coarse residue deposition facility and fine residue deposition facility.

New surface infrastructure will be required to support the underground during development and production. This infrastructure includes, but is not limited to:

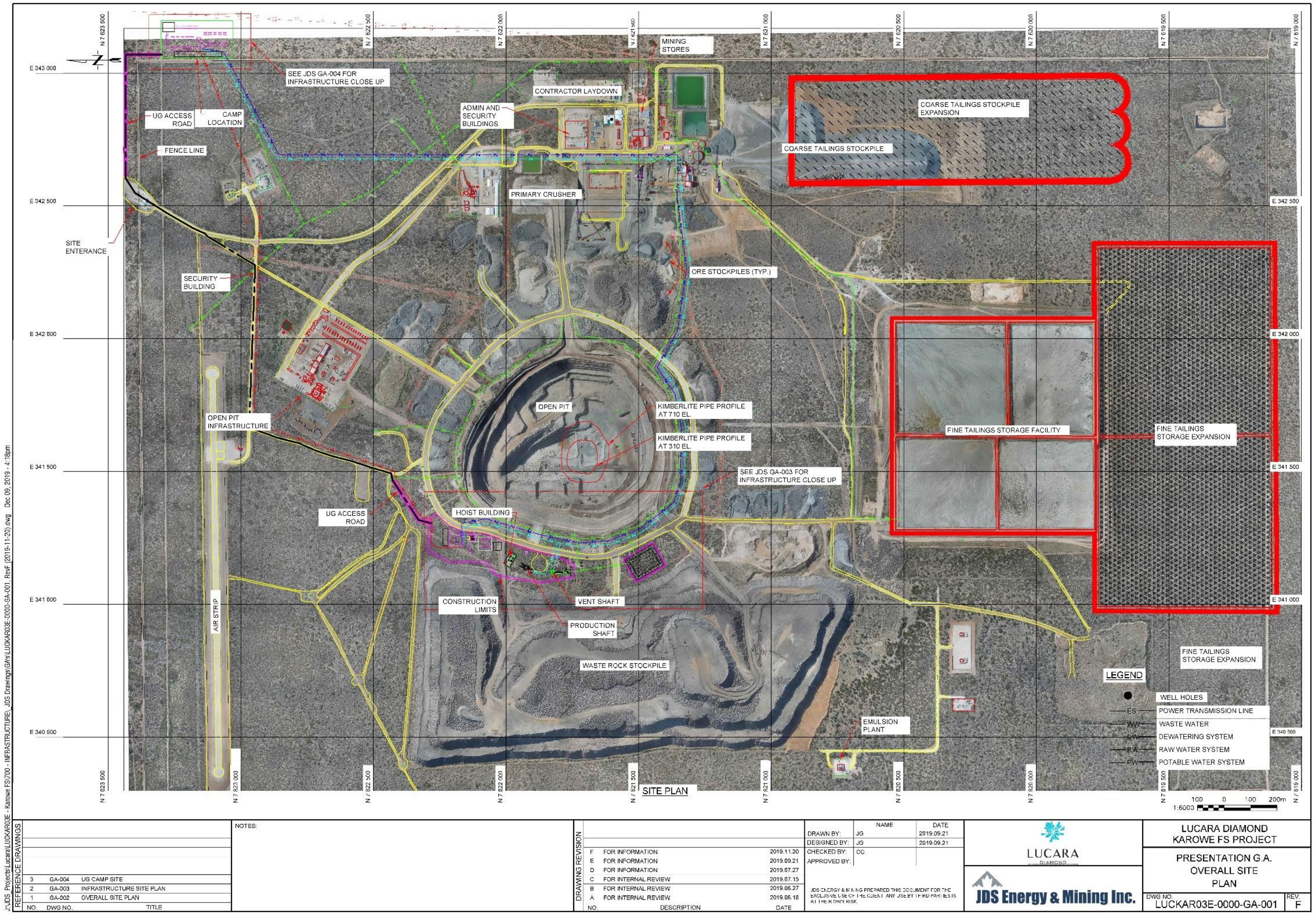
- New power supply line feeding the project site, including a new substation at the connection point to the grid supply;
- Underground area surface substation and power distribution from the existing site substation;
- Camp complex to support the construction workforce;
- Temporary power supply to support construction;
- Change house;
- Infrastructure pads and roadways;
- Surface sediment pond for managing underground dewatering; and
- Buildings and facilities to support the operation including:
    - Underground office complex;
    - Lamp and line out rooms;
    - Training and meeting rooms; and
    - Local first aid room.

18.1 General Site Arrangement

The site layout has been designed to minimize any additional land disturbance, minimize impact on existing operations during construction, provide security-controlled site access, minimize construction costs and optimize operational efficiency. The existing infrastructure will be utilized to the maximum extent possible.

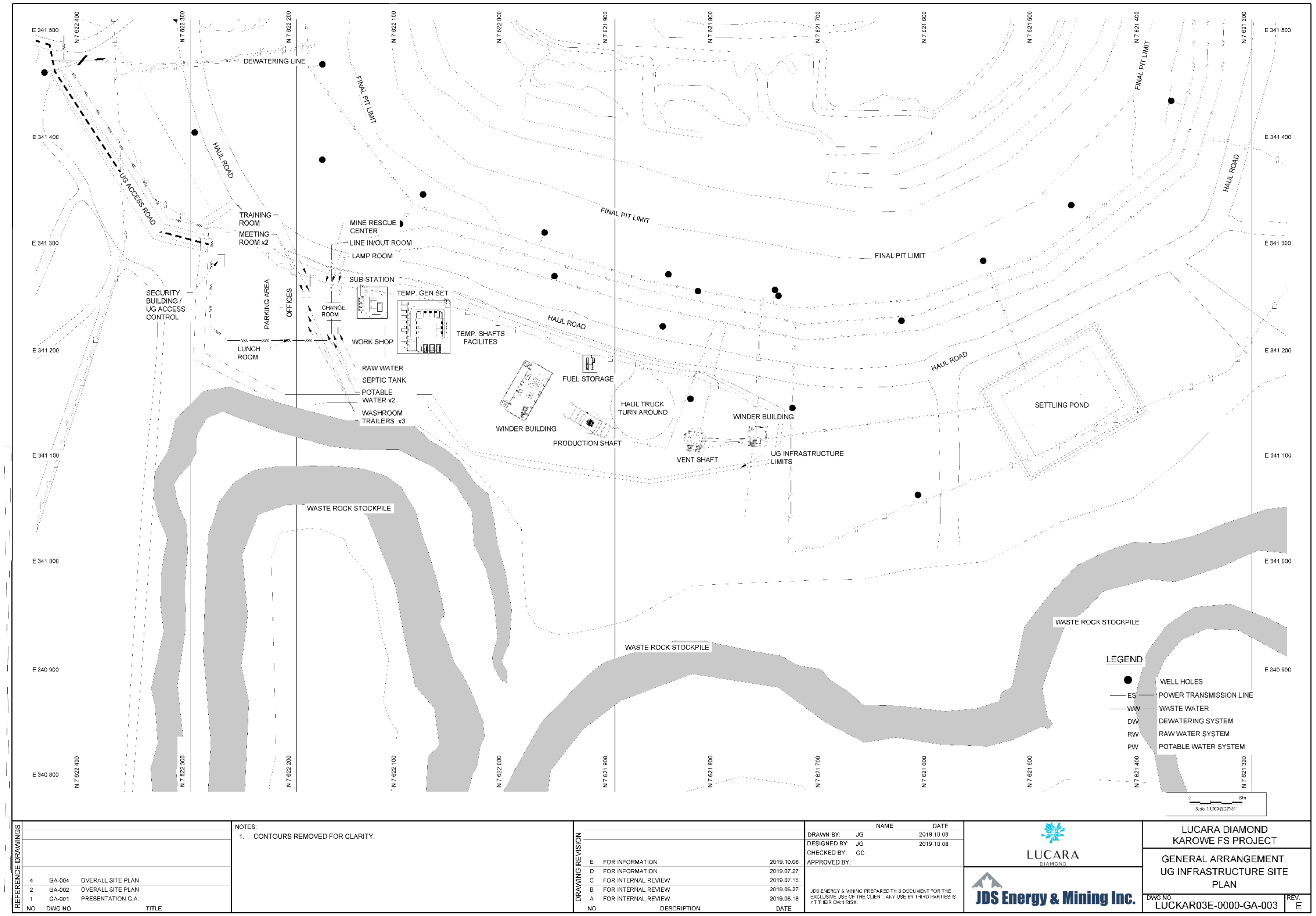
The Project site overall layout is provided in Figure 18-1. The existing fence line for the site is identified by the black outine surrounding the project site and is labeled in Figure 18-1. The current mine lease extends past the existing fence line, and all proposed infrastructure sits within the current mine lease. The main area of the underground infrastructure is shown in Figure 18-2.

Figure 18-1: Karowe Project Site General Layout



Source: JDS (2019)

Figure 18-2: Underground Infrastructure Layout



Source: JDS (2019)

18.2 Site Access

18.2.1 Current Access

The site is currently accessible by land via gravel road from the town of Letlhakane. There is also a private 1,480 m x 18 m airstrip, which is suitable for light aircraft, located at the project site which is used for product shipment and occasional site visits. Site access from the main access road is controlled by a main gate located in the north east of the project site. Vehicles (including buses, freight and supply vehicles) pass through the main gate before continuing along the mine access road to a parking and staging area near the Personnel Control Centre (PCC), where access to the main site is restricted.

18.2.2 Underground Project Road Access

Access to the Underground Project (UGP) will be through the main gate. Once through the gate, a turn off the mine access road will lead to an existing gravel road which currently accesses the airstrip. A vehicle access point will be located at the turnoff to control entry into the UGP area. UGP access will continue to follow existing site roads, however, approximately 225 m of new gravel road will be constructed to tie in the existing roads to the UGP pad. Minor upgrades to the existing roads may be required to ensure 8 m of useable width to allow for two-way traffic. Due to the flat topography of the project site, upgrades would consist of clearing and grubbing along the right of way and minor road surfacing as required from locally sourced calcrete, which is prolific across the site.

18.3 Buildings and Structures

As the mine is currently in operation, there are a number of existing facilities on the project site that will continue to be used during the construction and operation of the UGP. Where required, additional facilities, adjacent to the UGP area, will be developed to better support the operation. These are described in the following sections.

18.3.1 Office Buildings

In addition to the main office block, additional modular offices will be constructed near the shaft location to support the construction and operation of the UGP. Five 11.6 m x 3 m buildings, each with four offices, will be constructed to support the project specific workforce.

18.3.2 Training & Meeting Rooms

Modular buildings will be established next to the office blocks comprised of two 6 m x 3 m meeting rooms and one 11.6 m x 7 m training facility.

18.3.3 Maintenance and Storage

There is an existing open-air, covered maintenance facility located at the open pit infrastructure pad, which will be used to service the maintenance requirements on surface for the UGP. A new, covered facility with concrete floors and seacan / modular storage will be built near the UGP for local storage and minor maintenance requirements.

18.3.4 Lamp Room / Line Out Facility

Two modular 11.6 m x 3 m buildings will be constructed as lamp room and line out facility.

The lamp room will be fitted with storage racks to allow for charging of the lamps when not in use. Underground employees will collect their lamps from the associated racking. They will then proceed to the line out room to sign in on a personnel tracking board, prior to then walking to the shaft for access into the underground.

18.3.5 First Aid

The UGP will be serviced by the existing medical facility near the main office block. A local Mine Rescue Centre (MRC), will be constructed from a modular 11.6 m x 3 m building and located next to the lamp room. This facility will provide immediate first aid support to the UGP area and will provide surface storage for rescue and safety equipment for the underground.

18.3.6 Change House

A new, air-conditioned, change house facility will be constructed adjacent to the shaft area. The change house will service up to 100 people, or 50 per shift, with 80:20 male / female breakdown and includes a separated laundry and shower area. Clean and dirty lockers and storage will be included in the building, with a covered area at the entry with baskets for staff to remove the dirtiest outer layers prior to entry into the building. A general layout of the facility is shown in Figure 18-3.

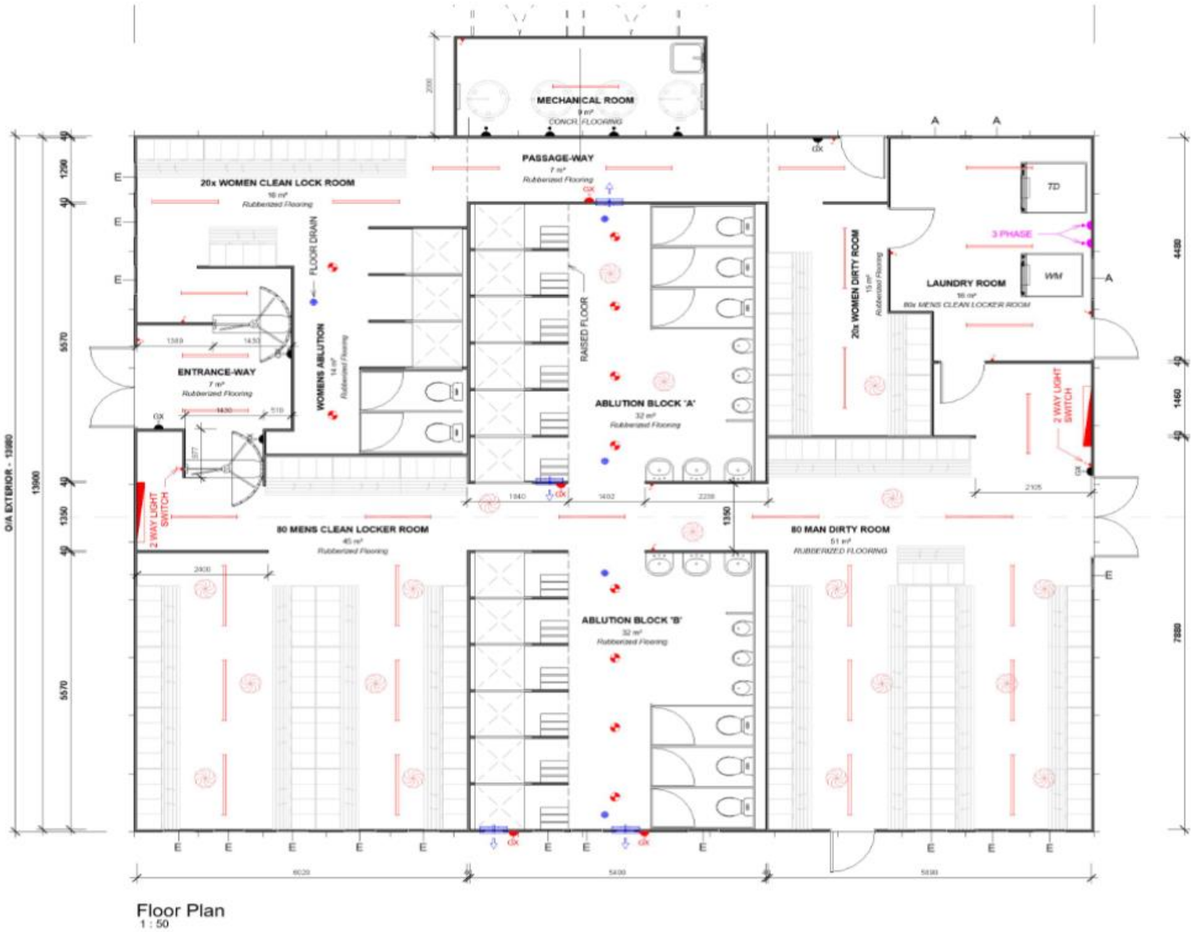
18.3.7 Security

As described in Section 18.2, the first point of access will be through the existing main security gates. A new vehicle check point will be constructed at the turnoff from the main site road to the underground access road. Chain link type fence will be constructed along the south / eastern side of the access road to prevent access into the open pit area.

Access into the UGP infrastructure area will be restricted through a high security access building, similar to the existing control point near the main office block. Multiple turnstiles will control foot access, with scan card and biometric validation required to pass. Vehicles will enter through a controlled vehicle gate and will be subject to search. All personnel and vehicles may be subject to search upon entry and exit of the facility. The high security access building will have space for UGP specific security personnel, to support the local security requirements; however, CCTV monitoring will be through the existing main security facility as part of the overall site security.

The UGP infrastructure area will be surrounded by high security fencing to prevent access to the area from the rest of the project site.

Figure 18-3: Change House



Source: Speed Space (2019)

18.3.8 Fuel Storage

There is an existing bulk fuel storage facility that will continue to be used for the UGP. A local 5,000 L fuel storage tank will be located near the shaft entrance as a filling point for totes that will be used to bring fuel down the shaft to support the underground operation.

18.3.9 Explosives Storage

The site currently has an emulsion plant located southwest of the waste dumps. The emulsion silos have a 200-ton capacity. In addition, there are two explosive magazines, each with 7,750 kg capacity. This facility will be continued to be used to service the UGP.

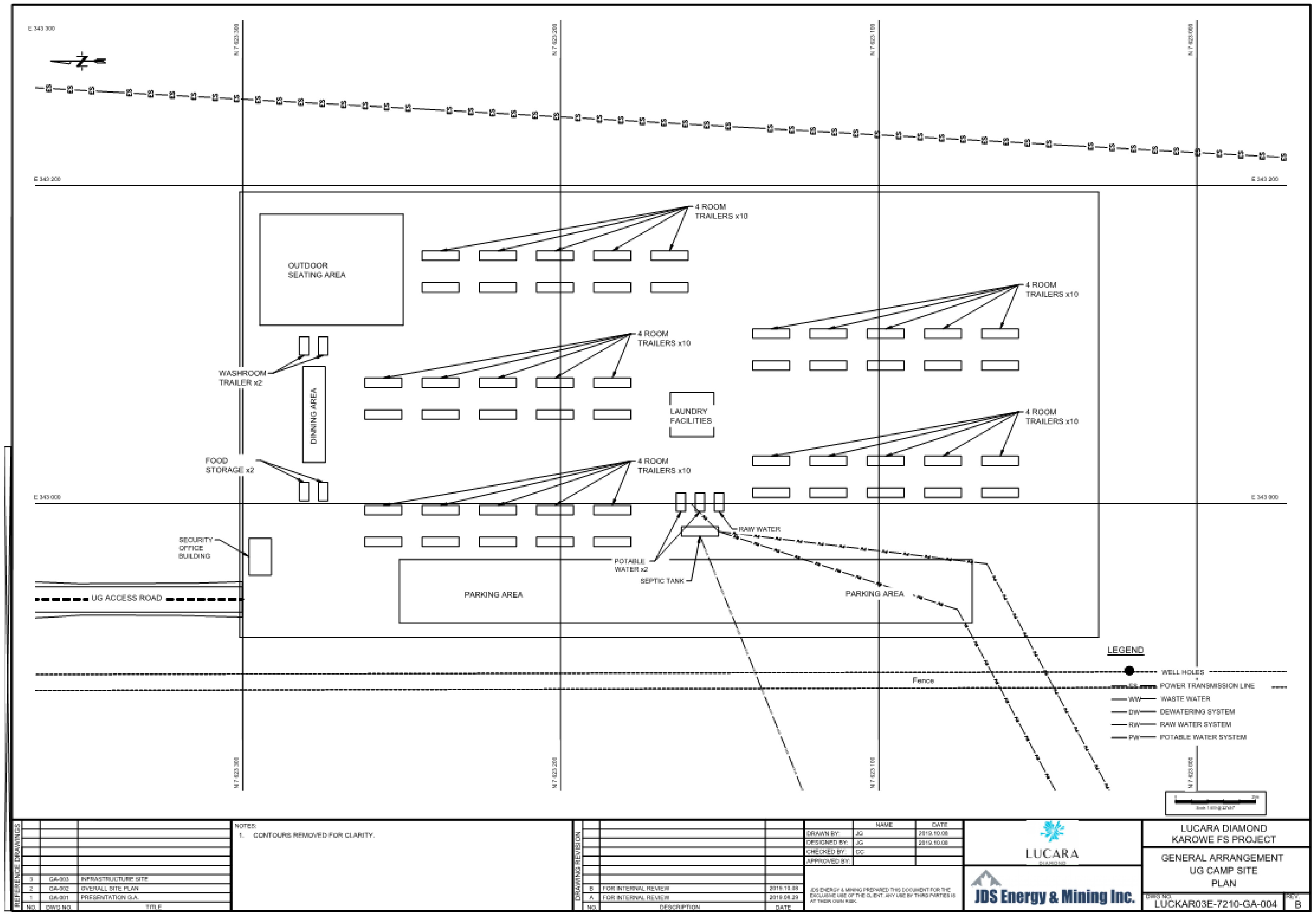
18.3.10 Camp

A temporary camp, for use during the construction of the UGP will be located near the northwest corner of the mine site. The camp will be sized for 200 people, made up of 50 x 4 room modular trailers. Each room will have individual washroom / shower access. An overall layout of the camp is shown in Figure 18-4.

The camp site will have security and office trailer which will act as an access check point. Security cards for the mine site will be issued to contractors from the camp security office, to reduce the demand on the main site access. The camp area will be surround by chain link fence to control access to the area.

The camp will have a dining facility, with space allocated for leisure and socializing. Next to the dining facility will be a lapaa area and outdoor eating space. There will be a centralized laundry facility, with laundry service provided to the contractors.

Figure 18-4: Underground Camp Site Plan



Source: JDS (2019)

18.4 Power

18.4.1 Bulk Power Supply

The Karowe UG operations will require additional bulk power with an estimated peak demand of 20 to 25 MVA, exceeding the existing contracted NMD of 12 MVA by around 13 MVA by 2025.

Electrical power to the plant will be supplied from the Botswana Power Corporation (BPC) Letlhakane 400 / 220 kV substation. A new 132 kV, 29 km transmission line will be constructed from the Letlhakane substation to a new 132 / 11 kV substation located within the premises of the mine.

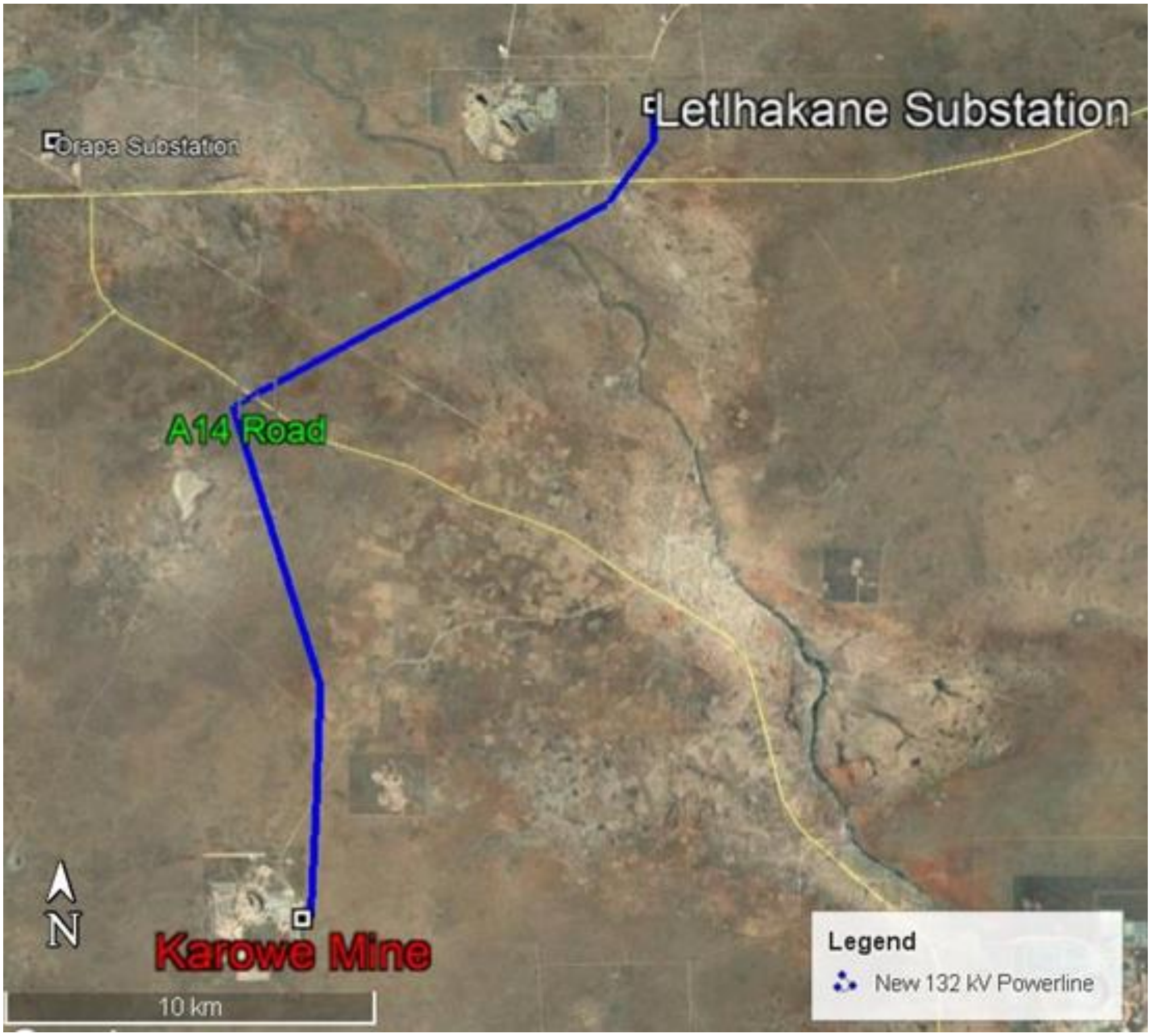
The existing Letlhakane 400 / 220 kV consists of a 400-kV yard and a 220-kV yard with a firm transformation capacity of 125 MVA. The existing substation will be extended to accommodate a new 132 kV switchyard with a firm transformation of 40 MVA. The new 132 kV yard at the Letlhakane substation will be a conventional open-terminal, air-insulated substation with two 40 MVA transformers installed to operate in parallel, to supply the full capacity required by the mine during a transformer failure.

The existing Karowe AK6 33 / 11 kV substation consists of a 33-kV busbar and an 11-kV busbar with transformation capacity of 30 MVA (i.e. two 15 MVA 33 / 11 kV transformers). The 33-kV busbar will be decommissioned when the integration of the new 132 / 11 kV switchyard into the existing 11 kV substation at the mine is completed. The new 132 kV yard at the AK6 substation will be a conventional open-terminal, air-insulated substation with a firm transformation capacity of 40 MVA (i.e. two 40 MVA 132 / 11 kV transformers). The interface between the existing 11 kV substation and the 40 MVA 132 / 11 kV transformer secondary 11 kV breakers will be through power cables rated for the full capacity of each transformer.

The new 132 kV powerline from the Letlhakane 400 / 220 kV substation to the new 132 / 11 kV substation at the mine will cross flat terrain running north of Karowe mine, adjacent to the existing BPC 33 kV powerline before crossing the A14 road. After crossing the road, it will turn northeast, following the BPC 400 kV powerline towards the source transmission substation as shown in Figure 18-5. The powerline will have rated transfer capacity of 90 MVA.

It is expected that the power will be available by April 2022.

Figure 18-5: Proposed 132 kV Powerline Route



Source: Royal HaskoningDHV (2019)

18.4.2 Underground Mine Bulk Power

18.4.2.1 General

The open pit mine currently in operation is served by an 11-kV substation located next to the BPC AK6 substation.

The substation supplies the following loads:

- Process plant;

- Open pit mining area;
- Surface infrastructure; and
- Pit dewatering bore-hole field.

The following voltage standards are applicable:

- Medium voltage (MV) 11 kV;
- Motors smaller than 250 V – 525 V;
- Motors smaller than 1,200 kW – 690 V;
- Motors larger than 1,200 kW – 4,125 V; and
- Lighting and small power 400 / 231 V.

18.4.2.2 New UG Mine Power Supply

The power supply to the new UG mine will be sourced from the existing 11 kV substation mentioned above. The 11-kV switchboard will be extended on each end of the board with new 12 kV rated circuit breakers.

A new "GOAT" conductor type overhead line on wooden poles will be constructed next to the current pit ring feed. The feed to the UG mine will also be in a ring feed configuration. Each leg of the ring will be able to support the full load of the mine.

The overhead line ring feed will terminate in a new 11 kV substation at the UG mine site. The substation will consist of a purpose-built e-house with 12 kV switchgear installed in the e-house. The switchgear will be Vacuum or Sulphur Hexafloride (SF_6) air insulated type switchgear.

The following loads will be supplied from the substation:

- A dual supply to the underground workings;
- A single supply to the primary shaft winder house;
- A single supply to the ventilation shaft;
- A single supply to the surface infrastructure (offices, change houses, etc.);
- A single feeder to the compressor house; and
- A single feeder to the shaft auxiliary transformer.

The medium voltage (MV) reticulation on site will be by 12.7 / 22 kV cross-linked poly-ethylene (XLPE) Type-A cables buried in trenches.

Transformers on the surface will be oil insulated air cooled (ONAN) type. For transformers smaller than 630 kVA mini substations with SF_6 MV switchgear will be used. For larger transformers, free-standing ONAN transformers will be used.

18.4.2.3 Power Factor Correction

The present power factor correction equipment in the main substation is designed for the current maximum demand of 12 MVA. The equipment will be upgraded to cater for the additional load due to the UG mine.

18.4.2.4 Standby Power

The requirement for standby power during a BPC power failure is estimated at 3.7 MW or 4.6 MVA. To cater for the load a standby diesel generator plant with a capacity of 5 MVA will be installed at the UG substation. The plant will consist of four 1,250 kVA 400 V generating sets. These are selected due to the easy availability of the units versus larger generators. Two 2.5 MVA step-up transformers will be installed to provide the 11 kV on the substation bus. Starting of the generator plant will be initiated manually during a power outage after the non-essential loads have been disconnected. Synchronization will be automatic after the units have started. It will be possible to back-synchronize the plant with BPC on the restoration of grid power to prevent lengthy start-up times.

18.4.2.5 Camp

A new contractor's camp will be established next to the project site. The total load of the camp is estimated at 500 kVA. A 639 kVA 11.4 kV mini substation will be installed at the camp.

Power supply to the camp will be by a "HARE" conductor type overhead line on wooden poles. The line will be a spur line taken from the new "GOAT" mine feed line.

Several distribution boards will be installed on the campsite, from where the individual units will be supplied.

18.4.2.6 Construction Power

Construction power is expected to increase up to a maximum of 13 MVA. This will be required for the first two years of construction. A rental diesel generator plant will be constructed to supply the contraction power requirements.

A total of fifteen 1,250 kVA generator units will be installed in groups of three, and each group will feed a 6.3 MVA transformer. Initially, one group of three units will be installed, and as the load requirements increase additional groups of three generators will be installed.

18.5 Water

18.5.1 Water Supply

Filtered and potable water is currently provided to site from a centralized water treatment plant near the thickener. The water treatment plant is comprised of the following water treatment processes: filtration, reverse osmosis and potable water treatment. The water supply to the treatment plant is provided by the raw water tank, which is fed by a pipeline from the open pit dewatering ring. The plant was expanded in 2018 and has the following production capacities on a daily basis:

- Filtration – 1,848 m^3/day;
- Reverse Osmosis ‐ 240 m^3/day; and

- Potable Water – 40 m^3/day.

In general, the plant operates below its design capacity; however, in order to accommodate the additional requirements of the UGP, the plant will be expanded to the following capacities.

- Filtration – 2,472 m^3/day;
- Reverse Osmosis – 360 m^3/day; and
- Potable Water – 60 m^3/day.

Filtered and potable water will be distributed to the UG area and the camp site via buried pipelines. Potable water and filtered water storage tanks will be located at the UG area near the change house. Additional water storage tanks will be located at the camp site to provide storage capacity in the individual areas.

18.5.2 Sewage Treatment

Sewage generated on site is currently collected in localized septic tanks near the buildings and wash car facilities. Sewage is then pumped from the septic tanks to a centralized sewage treatment plant located near the main access gate via underground piping systems.

The existing treatment plant is an activated sludge process where air or oxygen is forced into the sewage liquor to develop a biological floc which reduces the organic content of the sewage. The activated sludge is then sent through a clarifier, settled sludge is then returned back into the system and clear effluent is then treated with chlorine prior to release. The current facility is sized to process 52 m^3/day (day shift only) and is currently operating at approximately 50% capacity.

In order to accommodate the additional demands of the camp and change house, the sewage treatment plant will be expanded by 50%, to process 77 m^3/day (day shift only). Local septic tanks will be installed the near the change house and at the camp site, providing local sewage collection. Sewage will then be pumped to the sewage treatment facility via buried piping.

18.6 Surface Water Management

18.6.1 Underground Dewatering Surface Water Management

Water from the underground will be pumped to surface, where it will be sent to a sediment pond located to the south of the vent shaft. The sediment pond is designed to hold a 1 in 100-year, four-day rain event when pumped from the underground. Water will be pumped from the sediment pond, into the existing dewatering ring, where it will be sent to either the raw water tank or the existing supply line.

18.6.2 Open Pit Storm Water Paddocks

In order to minimize the volume of water reporting to the underground during a heavy rain event, paddocks will be constructed along the ramps in the open pit at the end of the open pit mine life. The paddocks will collect water off the pit walls and ramps during the rain event, preventing the water from immediately reporting to the underground. The water collected during the rain event will percolate through the waste rock berms over time and will be managed by the underground dewatering system after the peak flows generated by the rain event have dissipated.

18.7 Waste Rock Management

Waste rock generated during mining is currently placed in the Waste Rock Storage Facility (WRSF), located to the west of the open pit and planned underground facility. The WRSF is currently divided into three storage areas, the Orapa waste dump to the north, Centre waste dump and Khwee waste dumps to the south. Waste rock generated during the construction of the shafts and development of the underground will be placed in the same facility. Due to the relatively small volumes of waste rock generated by the UG, no re-design or expansion of the existing facility is anticipated.

18.8 Residue Storage Facilities

18.8.1 Introduction

Contained within this section are the feasibility designs for the tailings storage facilities referred to as the Coarse Residue Deposit (CRD) and the slimes dams called the Fine Residue Deposits (FRD).

18.8.2 Design Criteria

The design criteria below represent the incremental requirements from the start of 2020 and does not include tailings deposited in the facilities to date.

Table 18-1: CRD and FRD Design Criteria

Criteria	Description	Units	Design	Source
Life Required		Yrs	21	Lucara
ROM		Mt	57	Lucara
Recovery		%	5	Lucara
Volume of CRD and FRD Generated		Mt	54	Lucara
Coarse Residue Deposits (CRD)				
Tonnes to Facility		Mt	24.4	Lucara
Production		t/mth	97,000	Lucara
Density		t/ m^3	1.7	Lucara/RH Report
Volume		m^3/mth	57,000	Lucara
Volume for life		Mm^3	14.3	Lucara
Maximum Height		M	34	Lucara - Survey
Slope Angle			1:1.5	Lucara - Survey
Estimated Area Required		Ha	47	KP
Fine Residue Deposit (FRD)				
Tonnes to Facility		Mt	29.8	Lucara
Production		t/mth	118,000	Lucara
Dry Density		t/ m^3	1.2	KP
Cohesion		kPa	10	Lucara
Permeability		m/s	1×10^{-8}	Lucara

Criteria	Description	Units	Design	Source
Slurry Density		t/ m^3	1.3 – 1.4	Lucara
Volume		m^3/mth	98 487	Lucara
Volume for Life		Mm^3	24.8	Lucara
Maximum Height		m	26	KP
Impoundment Wall Inside Slope		Ratio (V:H)	1:1.5	KP
Impoundment Wall Outside Slope		Ratio (V:H)	1:2	KP
Crest Width		m	10	KP
Estimated Area Required		ha	175	KP

Source: Knight Piésold (2019)

18.8.3 Geotechnical Investigation

A geotechnical investigation was performed under the estimated footprint area of both the FRD and CRD. In general, the investigation confirmed a stratigraphy that is typical for the region with no unusual or weak features noted. In addition, the materials within the test pits were sampled and tested. This information was used for the overall feasibility design of both the CRD and FRD respectively (Knight Piésold, 2019).

18.8.4 Coarse Residue Deposit

The coarse residue will form part of the current the CRD facility and will expand south to accommodate this material.

The existing CRD facility utilizes a single conveyor system. A second conveyor system is currently being added to the facility. In order to optimize the storage to space ratio, it is proposed that a three-leg conveyor system be implemented. The existing 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. The second conveyor system will deposit the second leg and a lower advancement rate to ensure that the second and third leg complete their respective legs simultaneously.

In addition to accommodating tonnages to 2041, the proposed design of the CRD will remain within the mine lease boundary and will not encroach on the existing and proposed landfill sites.

Taking the above into consideration, a CRD design was developed and is summarised in Table 18-2. Refer to Figure 18-6 for the proposed layout and section of the CRD.

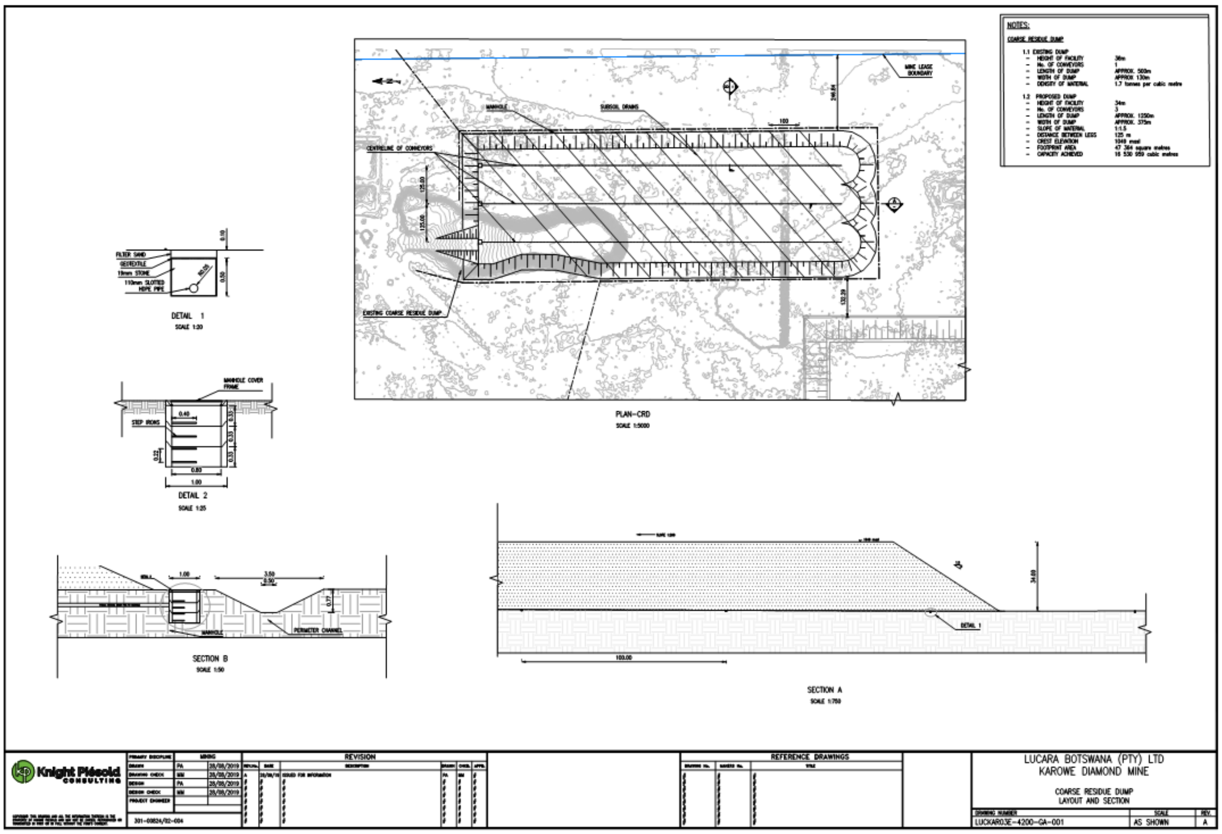
Table 18-2: Summary of Proposed CRD Facility Design Characteristics

Parameter	Units	Value
Design Life	Years	21
Total Storage Required	Mm^3	14.3
Total Storage Achieved	Mm^3	16.5
Footprint Area	km^2	47.4
Crest Elevation	masl	1,049
Height of Facility	m	34

Parameter	Units	Value
Number of Conveyor legs		3
Distance between conveyors (centre to centre)	m	125
Side slope of facility		1:1.5
Distance from Mine lease area	m	Approx. 246
Distance from landfill sites	m	Approx. 215

Source: Knight Piésold (2019)

Figure 18-6: Proposed Coarse Residue Deposit



Source: Knight Piésold (2019)

18.8.5 Fine Residue Deposit

The current deposition method of the fine residue on site is to place the material behind a waste rock impoundment wall. The current facility is divided into four paddocks, and the impoundment walls are raised in phases to ensure there is sufficient capacity for fine residue deposition and to maintain the legally required freeboard on the facility. A spigot operation is used to deposit the slurry into the active paddock and a pool forms towards the centre of the facility. The water is pumped from this point directly back to the plant.

The required expansion of the current facility is restricted by a number of items. To the west of the current facility, expansion is limited by the site topsoil stockpile: to the south, the mine lease boundary; to the east, the site landfill and future CRD footprint; and the pit to the north. Several options were modelled, and a trade-off study was conducted to determine the best option to accommodate the fine residue storage required.

In addition to the location constraints identified above, the ratio of waste rock to storage, which would provide a relative comparison of construction costs, was also used the deciding factor in determining which option would be further developed. The proposed options considered the possible extension of the existing facility (Phase 1) and the proposal of new FRD's (Phase 2 and Phase 3).

To evaluate which option would be best, a Multi-Criteria Analysis (MCA) was completed to determine which fine residue deposit option would be further developed. The MCA results are represented in Table 18-3 below.

Table 18-3: MCA Results

	Option A	Option B	Option C	Option D	Option E	Option F
Waste rock % of Fine Residue Volume	36%	48%	15%	23%	20%	46%
Rank	3	1	6	4	5	2
Score/40	20.0	6.6	40.0	26.6	33.3	13.3
Encroaches Topsoil Dump	No	No	No	No	No	Yes
Score/20	20	20	20	20	20	0
Encroaches Proposed Landfill Site	No	No	No	No	No	Yes
Score/20	20	20	20	20	20	0
Extends beyond Mine Lease Area	No	No	Yes	Yes	No	No
Score/20	20	20	0	0	20	20
Score/100	**80.0**	**66.6**	**80.0**	**66.6**	**93.3**	**33.3**

Source: Knight Piésold (2019)

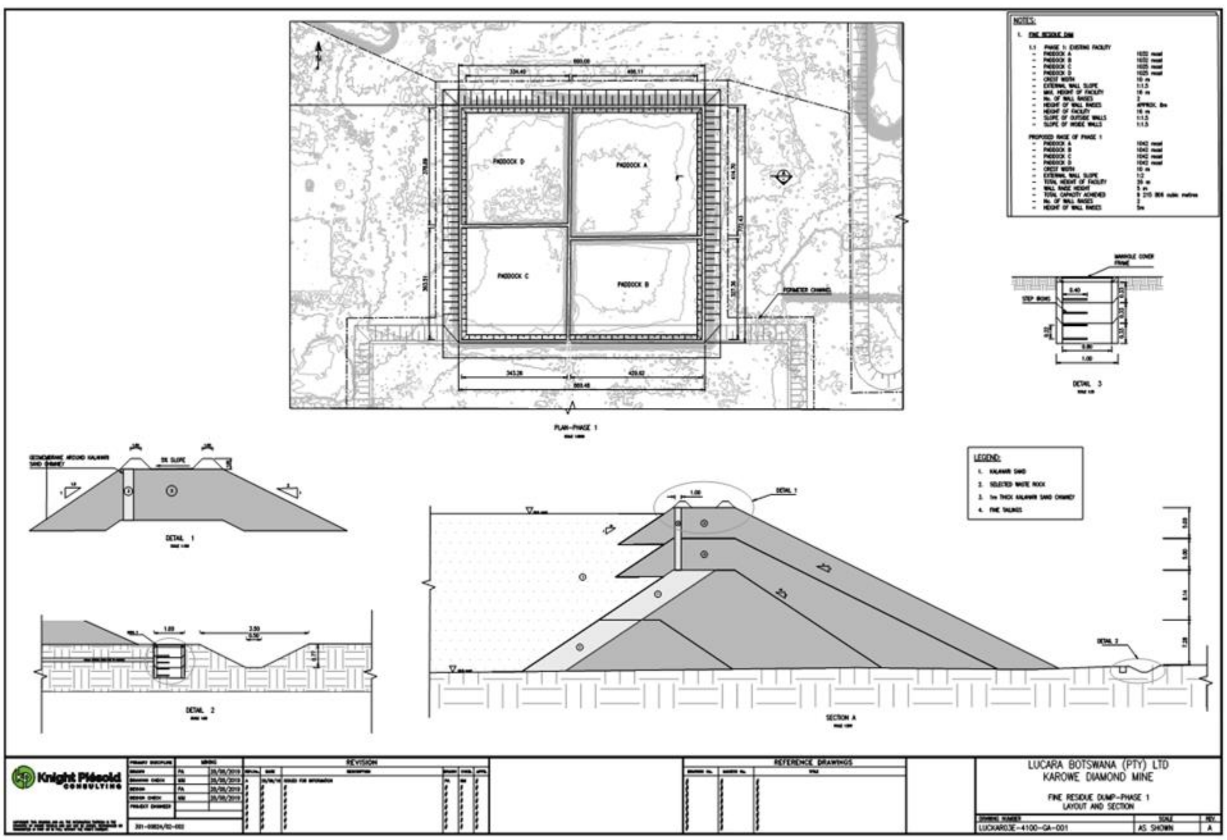
The results for the MCA yielded that Option E would be most favourable. Option E was then developed further with the final design shown in Figure 18-7. The design characteristics are summarized in Table 18-4. Refer to Figure 18-7 and Figure 18-8 for the proposed layouts and sections of the FRD Option E.

Table 18-4: Summary of Proposed FRD Design Characteristics (Option E)

	Units	Value
Design life from 2020	Years	21
Total storage required from January 2020	Mm^3	24.8
Total storage achieved from January 2020	Mm^3	27.3
Crest elevation	masl	Phase 1: 1,042 Phase 2: 1,042
Height of facility	m	Phase 1: 26 Phase 2: 25
Volume of waste rock required	Mm^3	6.2

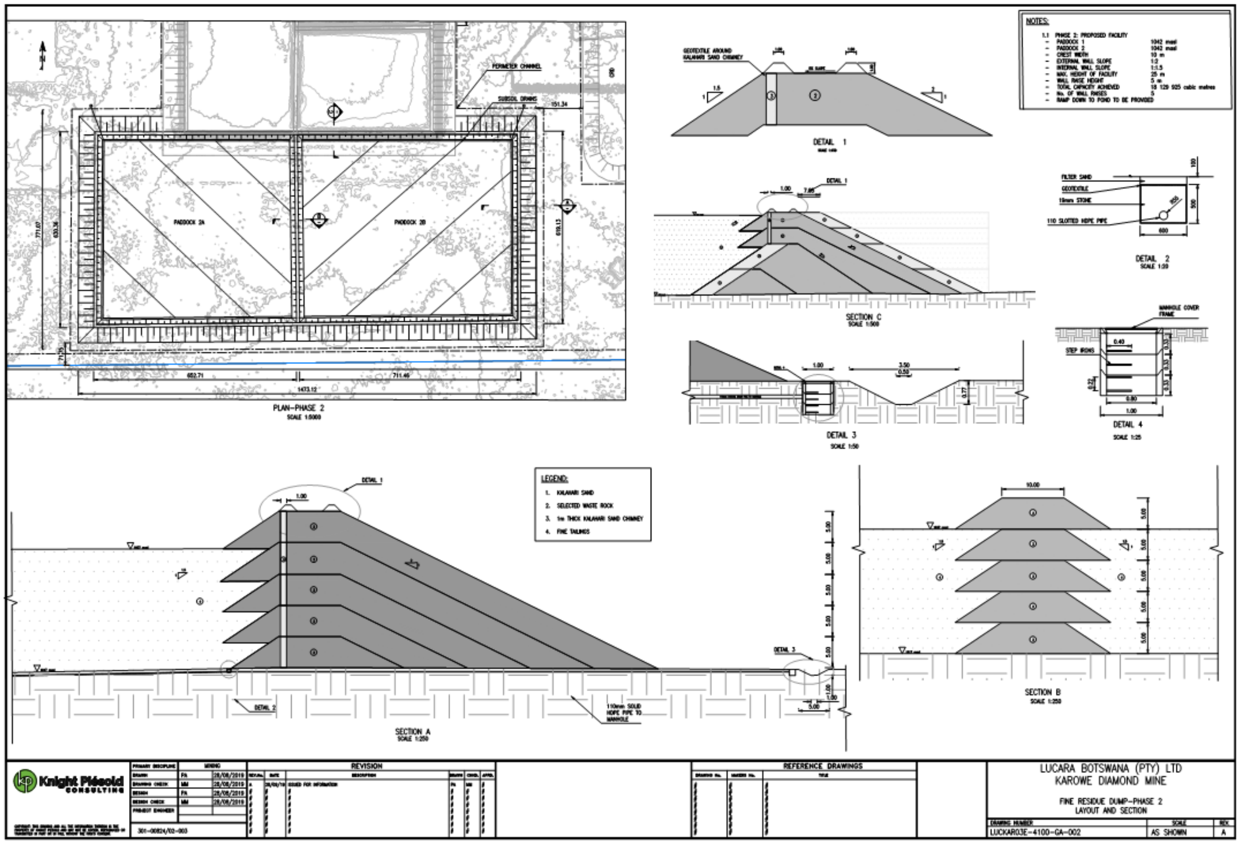
Source: Knight Piésold (2019)

Figure 18-7: Fine Residue Dump - Phase 1 Layout and Section



Source: Knight Piésold (2019)

Figure 18-8: Fine Residue Dump - Phase 2 Layout and Section



Source: Knight Piésold (2019)

The design of the FRD Option E resulted in a facility with the following features:

- *Phase 1:* The impoundment wall will be raised to a height of 1,042 from the original 1,032 design height. This will be done in two 5 m raises. The four paddocks will be maintained on the facility; and
- *Phase 2:* The impoundment wall will be raised to a height of 1,042. This will be done in five 5 m raises. Phase 2 will be divided into two paddocks to make wall raising and operations of the facility easier.

The impoundment walls will have 5 m centre wall raises and are designed to have a 10 m wide crest to provide access on the facility. The downstream slope of the wall will be 1V:2H and the upstream slope will be 1V:1.5H. The wall should be constructed in 500 mm thick layers of selected waste rock, compacted to the site-specific developed standard. A one m wide Kalahari sand chimney will be installed on the inner crest of the wall to prevent piping through the waste rock.

The decant system for the facility will be a continuation of the current system being used on site. It consists of a surface pump with a suction pipe running into the water.

18.8.6 FRD Capacity Analysis

The storage capacities of both the FRD and CRD facilities were determined using Muk3D modelling software. In addition to the storage capacities, the required time to deposit the calculated volumes was also determined. These results are shown in Table 18-5 and Table 18-6.

Table 18-5: FRD Volumes Achieved and Time to Fill-Phase 1

Cell	Planned Berm Elevation (masl)	Height of Facility (m)	Volume Achieved (From January 2020) (m^3)	Months	Years	Total Period per Raise (Years)
Raise 1						
Paddock B	1,032	16	844,411	8.6	0.7	2.24
Paddock C	1,032	16	754,503	7.7	0.6	
Paddock D	1,032	16	845,667	8.6	0.7	
Raise 2						
Paddock A	1,037	21	838,775	8.5	0.7	2.29
Paddock B	1,037	21	657,662	6.7	0.6	
Paddock C	1,037	21	590,142	6.0	0.5	
Paddock D	1,037	21	617,565	6.3	0.5	
Raise 3						
Paddock A	1,042	26	831,620	8.4	0.7	2.27
Paddock B	1,042	26	645,934	6.6	0.5	
Paddock C	1,042	26	590,144	6.0	0.5	
Paddock D	1,042	26	614,652	6.2	0.5	
Total			7,831,075			7.8
Rate of Rise = 3.05 m/year						

Source: Knight Piésold (2019)

Table 18-6: FRD Volumes Achieved and Time to Fill-Phase 2

Cell	Planned Berm Elevation (masl)	Height of Facility (m)	Volume Achieved Starting in 2020 (m^3)	Months	Years	Total Period per Raise (Years)
Raise 1						
Paddock 2A	1,022	5	1,329,979	13.5		2.4
Paddock 2B	1,022	5	1,490,011	15.1		
Raise 2						
Paddock 2A	1,027	10	1,788,094	18.2	1.5	3.2
Paddock 2B	1,027	10	1,944,062	19.7	1.6	
Raise 3						
Paddock 2A	1,032	15	1,819,640	18.5	1.5	3.2
Paddock 2B	1,032	15	1,982,439	20.1	1.7	
Raise 4						
Paddock 2A	1,037	20	1,848,882	18.8	1.6	3.3
Paddock 2B	1,037	20	2,018,622	20.5	1.7	
Raise 5						

Cell	Planned Berm Elevation (masl)	Height of Facility (m)	Volume Achieved Starting in 2020 (m^3)	Months	Years	Total Period per Raise (Years)
Paddock 2A	1,042	25	1,864,417	18.9	1.6	3.3
Paddock 2B	1,042	25	2,043,779	20.8	1.7	
		Total	18,129,925			15.3
Rate of Rise = 1.50 m/year						

Source: Knight Piésold (2019)

The total waste rock required to construct the wall raises for Phase 1 and Phase 2 is 6,164,550 m^3.

18.8.7 Hazard Classification

Safety classification of the FRD facility, in accordance with the criteria in South African National Standards (SANS) 10286:1998 "Code of practice, Mine residue", is dependent upon the zone of influence of the facility. This is the area around the dam in which a failure would have the effect of causing loss of life, damage to property and pollution of the environment. The code prescribes the aims, principles and minimum requirements that apply to the classification procedure and the classification in turn gives rise to minimum requirements for investigation, design, construction, operation and decommissioning.

The boundary of the zone of influence is determined as follows (where h is the height of the facility at the point under consideration):

a) Upstream of any point on the perimeter, the lesser of a distance of 5 x h from the toe; and the distance to the point where the ground level exceeds h / 2 above the elevation of the toe at the point on the perimeter.

b) On the sides parallel to the ground slope – a distance of 10 x h from the toe.

c) Downstream of the lowest point on the perimeter – a distance of 100 x h up to a maximum of 6 km.

Based on the zone of influence as seen below, the facility is classified as high hazard facility. Although no residents live in the zone of influence, there is potential for the flow slide to cause harm to the mine plant and the pit to the north of the facility. Table 18-7 shows the SANS10286 hazard classification method with the zone of influence shown on the following page in Figure 18-9.

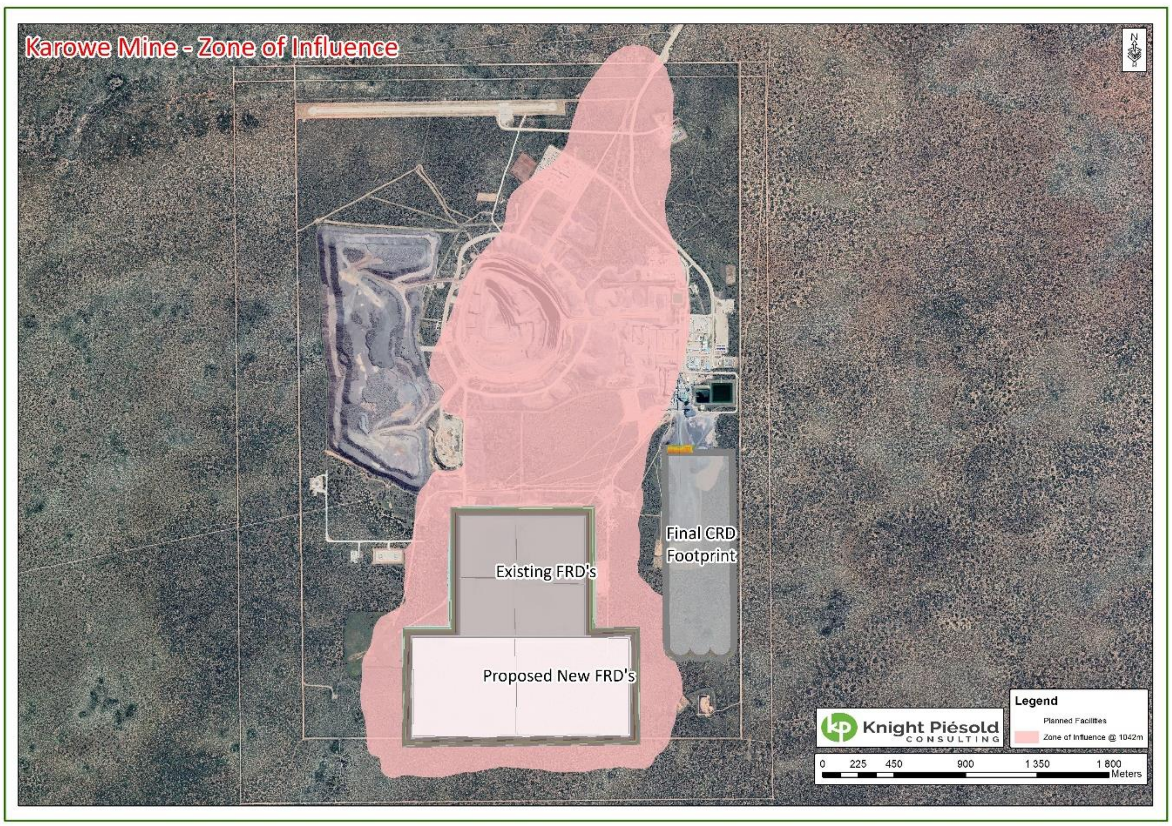
Table 18-7: SANS10286 Hazard Classification

Classification	Number of residents in zone of influence	Number of workers in zone of influence1	Value of third-party property in zone of influence2	Depth to underground mine workings3
High Hazard	>10	> 100	> R20million	< 50m
Medium Hazard	1 - 10	11 – 100	R2million – R20million	50m – 200m
Low Hazard	0	<10	R0million – R2million	> 200m

1. Not including workers employed solely for the purposes of operating the deposit.
2. Values are as per SANS 10286 1998
3. The potential for collapse of the residue deposit into the underground workings effectively extends the zone of influence to below ground level.

Source: Knight Piésold (2019)

Figure 18-9: Zone of influence by SANS 10286



Source: Knight Piésold (2019)

18.8.8 Stability Assessment

18.8.8.1 FRD Methodology

A preliminary stability analysis was done to assess the FRD facilities. The stability analysis should be updated with more comprehensive material testing for the detailed design stage. The following factors of safety (FoS) were used:

- Drained FoS minimum is 1.5;
- Undrained (Peak) FoS minimum is 1.3; and
- Seismic FoS minimum is 1.5.

Centreline wall lifting technique was used for the feasibility design whereby the centreline of the impoundment wall stays constant for each wall lift. Wall lift height was set at 5 m per lift. To determine the required slope angle for the waste rock wall impoundment the slope was increased from 1:1.5 until the required FoS was achieved. The final slopes were determined to be 1:1.5 upstream slope with a 1:2 downstream slope.

18.8.8.2 CRD Methodology

The following factors of safety were used:

- Global FoS minimum is 1.3; and
- Global seismic minimum is at 1.1.

A global failure is one which affects the stacker and/or conveyer line on the CRD. The material is stacked at the angle of repose (approximately 1:1.5) and will therefore have a shallow slip surface failure FoS of approximately 1. The water content of the CRD plus the predominantly gravel size meant that no pore pressure effects were modelled. The conveyor is positioned approximately 22 m away from the advancing face. An additional stability analysis was performed with an assumed spreader loading.

18.8.8.3 Stability Results

The stability analyses of the final selected slopes yielded the results presented in Table 18-8.

Table 18-8: FRD Phase 1 – Summary of Stability Results

FRD - Phase 1	Lift	Drained	Undrained (Peak ratio)	Drained Pseudo-Static
		Minimum Required		
		1.5	1.3	1.1
Downstream at 1:2	Lift 1 - 1037 masl	1.6	1.6	1.2
	Lift 2 - 1042 masl	1.6	1.6	1.2
Upstream at 1:1.5	Lift 1 - 1037 masl	1.8	1.1	1.3
	Lift 2 - 1042 masl	2.4	1.5	1.5

*Note: red denotes failure obtain the recommended FOS.
Source: Knight Piésold (2019)

The upstream FoS may be below the required minimum due to the wall being constructed on loose fine residue. It is recommended that a waste rock pioneer layer is placed on the fine residue before the construction of the lift or a suitable engineered alternative design which should be performed during detailed design. These results are based on the assumed and/or tested material parameters from limited samples. During the detailed design phase these values should be confirmed with extensive testing.

Table 18-9: FRD Phase 2 – Summary of Stability Results

FRD - Phase 2	Lift	Drained	Undrained (Peak ratio)	Drained Pseudo-Static
		Minimum Required		
		1.5	1.3	1.1
First Wall	Wall - 1022 masl	2.5	2.3	1.6
Downstream at 1:2	Lift 1 - 1027 masl	1.8	1.7	1.3
	Lift 2 - 1032 masl	1.6	1.6	1.3
	Lift 3 - 1037 masl	1.6	1.6	1.2
	Lift 4 - 1042 masl	1.6	1.6	1.2

Source: Knight Piésold (2019)

Table 18-10: CRD – Summary of Stability Results

CRD	Lift	Drained	Drained Pseudo-Static
		Minimum Required	
		1.3	1.1
Without Spreader Loading	Wall - 1022 masl	1.5	1.1
With Spreader Loading	Lift 1 - 1027 masl	1.5	1.1

Source: Knight Piésold (2019)

18.8.9 Storm Water Management

A storm water management plan was developed to mitigate the risk of the site becoming inoperable during major storm events due to runoff from the CRD and the FRD flowing through the site. Typically this involves diverting non-contact water (natural runoff upstream of a site that has not come into contact with mining related surfaces) away from the site, and controlling the flow of contact water (runoff within the site that that has come into contact with mining related surfaces) and routing it to a storage facility or controlled discharge point.

18.8.10 Water Balance

In order to demonstrate that the capacity of the FRD's is sufficient in containing the dirty water generated from fine residue deposition and direct precipitation, a daily time-step volumetric water balance was modelled in MS Excel™. The water balance was modelled according to the proposed deposition strategy and considers losses due to entrainment, seepage, evaporation, and re-use. The deposition will be cycled between the different paddocks, with only one paddock being actively deposited on at a time.

Based on the deposition strategy, and the daily time-step water balance, the minimum freeboard for each paddock is presented in Table 18-11. Based on South African guideline GN 704, the minimum operating freeboard for a facility that stores contact water is 0.80 m above the full supply level, which is achieved by all the paddocks. The standard code of practice document SABS 02861: 1998 gives guidelines for the maximum storm volume that the facility should contain over and above the maximum operating capacity of the facility. Based on this guideline, the facility should be able to contain the 24-hour storm with a recurrence interval of 1 in 100 year and still maintain a freeboard of 0.50 m.

This 100-year storm event depth falls directly onto the catchment and is assumed to fully contribute to the pond volume. It was found that if the storm were to occur while any of the paddocks were already at their minimum operating freeboard, they would still have more than 0.50 m freeboard remaining, which is compliant with both guidelines.

Table 18-11: Minimum Operating Freeboard Achieved per Paddock

Paddock	1A	1B	1C	1D	2A	2B
Minimum Operating Freeboard (m)	0.82	0.85	0.80	0.83	0.84	0.98

Source: Knight Piésold (2019)

In summary, the designed facility will comply with both SANS10286 and GN704 of the water act. The minimum freeboard requirement of one metre is therefore sufficient.

18.8.11 Conclusion

In summary the FRD and CRD facilities can be expanded to accommodate the proposed underground mining extension.

19 Market Studies and Contracts

This section is contributed by Lucara under the oversight of Dr. John Armstrong. The information documented herein was extracted and summarized from Nowicki et al. (2018) and updated where relevant to August 2019.

Under the terms and conditions contained within ML 2008/6L, Boteti will hold open tenders for sale of diamonds in Botswana. In the period 2012 to the end of 2014, dual viewing of goods was held in Antwerp and Gaborone with the final tender closing in Antwerp. Since January 2015, all diamond tender viewings and sales have taken place in Lucara's dedicated sales and marketing office within the Diamond Technology Park, Gaborone. In Q1 2018, Lucara acquired Clara Diamond Solutions ("Clara"). Clara, now a wholly owned subsidiary of Lucara, is developing a secure, digital sales platform that uses proprietary analytics together with cloud and blockchain technologies to modernize the existing diamond supply chain, driving efficiencies, unlocking value and ensuring diamond provenance from mine to finger. A portion of production from Karowe is now sold through the Clara platform.

Lucara manages a rough price book (>4000 price points) that generates a reserve price for each sales lot. Specials (+10.8 ct and coloured diamonds) are treated on an individual basis. The Government Diamond Valuator (GDV) also completes a valuation of the rough lots to be tendered and reserve prices are compared prior to tender or release to the Clara platform. The costs of the GDV are for the account of the Government. Royalty payments are calculated on the actual sales price for achieved during tenders and sales through the Clara platform.

19.1 Diamond Sales

Since 2012 over 2.5 Mcarats of combined North, Centre and South lobe diamonds have been sold for revenue of US$1.5 B (average price per carat of US$586/ct).

Sales lots are prepared for presentation to clients by Lucara Botswana staff in a modern, ultra-secure sorting facility. Sales parcels conform to industry standard size ranges and descriptions.

Karowe Mine production includes on a consistent basis a proportion of large, high value Type IIa diamonds and infrequent coloured diamonds (blue, pink, yellow). Diamonds such as these are very rare and command a special niche within the rough and polished markets.

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

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

Lucara adjusts its sales strategy to maximize client participation and achieve best possible revenue. In Q2 2018, Lucara moved to a blended tender process, whereby diamonds recovered in the sales cycle period are tendered and not held in inventory. A greater number of exceptional stones will be sold as part of RST's. This will decrease the inventory time for large, high value diamonds and will generate a smoother, more predictable revenue profile that better supports price guidance on a per sale basis.

In February 2018, the Company acquired Clara Diamond Solutions Corp. ("Clara"). Clara, a wholly owned subsidiary of Lucara, has developed a secure, digital sales platform that uses proprietary analytics together with cloud and blockchain technologies to modernize the existing diamond supply chain, driving efficiencies, unlocking value and ensuring diamond provenance from mine to finger. During 2018, Lucara commercialized Clara and conducted the first sale through the platform. During 2019, diamonds recovered between November 2018 and July 2019 were sold either in a blended sales tender or through the Clara digital sales platform. A selection of Karowe's production sized between 1 and 4 carats and of better quality were offered for sale on the Clara platform, as the platform matures additional production will be sold via Clara.

19.2 Client Base

Lucara has developed a strong, geographically diverse following of clients. Lucara has 713 registered clients, demonstrating a strong interest in the Karowe production. Attendance at tenders has increased to an average of 119 companies in the period of 2019 compared to 113 in 2018.

19.3 Rough Diamond Market Outlook

The overall rough and polished markets remain cautious and under price pressure due to a variety of macro economic, and supply and demand fundamentals remaining unbalanced. New rough producers that came online through 2016 and 2017 (Renard, Gahcho Kué, Liqhobong) achieved market prices for new production that have not met expectations as a result mainly of market conditions.

Current issues during 2018-2019 that are applying pressure to the rough market include:

- Demonetization in India
    - Has had an overall impact on the market but in terms of rough pricing the impact was not as significant with prices off mainly in poorer quality smaller goods.
- Uncertainty regarding China – USA trade agreements;
- Political unrest in Hong Kong; and
- The de-valued ruble.

Smalls and commercial goods came under rough diamond pricing pressure in 2018-2019, with price decrease on the order of 10-15%. Large, high-quality rough, came under pricing pressure beginning in early 2019. Precipitous decreases in demand and pricing of large, high-quality polished diamonds preceded and accompanied the weakness in large, high-quality rough. Lucara is advantageously placed in the market with a source of high value large diamonds, and therefore pricing and achieved average prices is sensitive to movement in the large high-quality goods.

Price adjustments for the +10.8 ct sizes have been required, with variances to 2018 on the order of 25-35%. Demand and flat to slightly positive improvements were observed in Q3 2019 in the large diamond sector.

A strong, expanding customer base, excellent participation in tenders, adoption of the Clara Platform, and a consistent production profile that is trending toward more higher-grade, South Lobe and EM/PK(S) production with consistent sorting and presentation of sales lots has generated a Lucara brand where the outlook is positive.

20 Environmental Studies, Permitting and Social or Community Impacts

20.1 Environmental Studies Completed to Date

20.1.1 Historical

Two pre-mining environmental studies were conducted for the Karowe Mine (formerly known as the AK6 project), namely an Environmental Impact Assessment (EIA) Study for AK6 (Geoflux, 2007) and Environmental Management Plan (EMP) for the AK6 Diamond Mine (SiVEST, 2010). As the responsible authority, the Botswana Department of Environmental Affairs approved both studies in 2008 and 2010, respectively. In terms of the Mining License (ML 2008/6L); Boteti Mining was granted common law surface rights over the entire mining license area and the access road for the duration of the mining lease.

20.1.2 Permitting

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

Permitting applications for the site's waste facilities (salvage yard, landfill, sewage plant and incinerator) initiated over the past three years, remain in process as of late 2019.

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

20.1.3 Capacity

During 2019, the Environment, Health, Safety & Community Relations (EHS & CR) Department was expanded in order to allow for more dedicated, separate capacity for the various functions. The department now houses dedicated health and safety, sustainability, environmental, stakeholder engagement as well as corporate social investment line functions.

20.1.4 Environmental Management

As required in terms of the Environmental Assessment Act of 2011, the 2016 EMP update sets out the mitigation measures and impact management / monitoring activities that KDM must undertake to maintain compliance during the current operational and later closure phase of the Project. Various reviews have recommended improvements in data gathering processes (Geoflux 2014; EBS 2017). The EMP was comprehensively updated in December 2018 with a risk assessment and included a review of potential impacts associated with the Underground Project. The specialist studies undertaken during the course of this update showed that with appropriate mitigation measures in place, no unacceptable impacts for the surrounding sensitive receptors should occur and non-compliances would be minimized. To maintain ongoing performance, all staff and long-term contractor induction and refresher training includes a sustainability component.

Specifically, the mine continues to monitor:

- Air quality by means of a dust bucket and emissions system sampling monitoring points located at key on- and off-site receptor points;
- Groundwater quality by means of and on- and off-site borehole monitoring systems as well as clean / dirty water control infrastructure on site, specifically monitoring potential seepage from the slimes dam;
- Surface water / storm water control infrastructure by way of infrastructure inspections to ensure the containment of mobilized pollutants in the event of spillages or significant rainfall events;
- Waste Management by means of a waste separation bin system and a lined, on-site landfill for non-hazardous waste; and
- Land disturbance and Environmental incidents by means of continuous inspections.

In all cases, monitoring samples are analyzed by independent third parties. Once mitigation measures are in place, dedicated monitoring campaigns will be used to test the efficiency of the mitigation measures in order to ensure compliance with the regulatory requirements. 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.

The updated EMP will be submitted for regulatory approval in early 2020 once all material aspects and impacts of the UGP have been finalized and evaluated.

20.1.5 Natural Setting

The Orapa-Letlhakane region is generally flat with a slight fall towards the north / northwest. Ground elevation ranges between 1,000 m in the south / southeast and 950 m further towards the northwest. Surface drainage is virtually non-existent, except for the dry Letlhakane River (fossil valley) which drains towards the Makgadikgadi pans.

The region is characterized by a semi-arid to arid climate with hot, wet summers and cold, dry winters. The highest temperatures are experienced during summer with maximum and minimum temperature averaging above 30°C and 20°C respectively. During the winter months, the average minimum temperature often falls below 10°C. The wind direction is quite variable, especially at low speeds (<7 knots). The majority of the high-speed winds blow to the west and west-northwest.

Rainfall in the Letlhakane area is temporary and spatially variable. Typically, most rainfall occurs between September and April, although some events have been recorded between May and August. The soils of the mining lease area comprise arenosols, luvisols and calsisols, covered in mopane tree and shrub, savannah with occasional grassy areas. Most of the surface flow tends to be localized to the numerous pans dotted throughout the region. The flat landscape is altered by the presence of silcrete / ferricrete hillocks in the east, the numerous pans, especially to the west and northwest, and manmade features of relatively high relief in the mining areas of Orapa and Letlhakane. These man-made 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 & Flora

The area of the Mining License (ML 2008/6L) falls within the range of most of Botswana's savanna species. However, due to intermittent grazing, occasional firewood gathering, as well as the mining operations nearby (all of which predate the establishment of the KDM), such species are sparse, and only occasional herbivores and bird species are sighted. None of the studies carried out as part of the EIA or the EMP and its updates indicate the presence of any rare, threatened or endangered animal species.

The area of the Mining License is covered by a mix of two vegetation types: mopane tree savanna on poorly drained soils with high clay content, and mopane shrub savanna on sand. None of the studies carried out as part of the EIA or the EMP and its updates indicate the presence of any endemic, rare, threatened or endangered plant species in the area.

20.1.7 Ground Water & Water Management

Groundwater studies in Orapa and Letlhakane region started at the same time as diamond mining operations in the Orapa-Letlhakane region in the early 1970's. Recent groundwater studies at Karowe by Exigo, based on monitoring wells, packer testing and actual dewatering well performance have provided a strong base of information, including hydrogeological models, for the FS. Groundwater in this region is extremely important for meeting demand (current and future) for mining, domestic supply and livestock watering.

Groundwater information is contained in Section 16.4 of this report. Analyses of ground water impacts are on-going.

All surface clean / dirty water management infrastructure is designed and maintained to prevent spillage of a 1:50 year rainfall event.

20.1.8 Fine Residue Deposits (FRD) Dam

The square-shaped FRD dam is located south of the open pit. The FRD dam is split into four equal sized compartments with a total footprint of approximately 146 ha. The four compartments are operated on a rotational basis (approximately three continuous months per annum for each) in order to minimize water losses. The FRD dam design adheres to South African National Standard (SANS) 10286; and due to the storage of water on the facility, all dam water management infrastructure and systems are built to manage flows arising a 1:50 rainfall event as per the requirements of GN 704 (July 4, 1999) of the South African National Water Act of 1998.

As stipulated in the EMP, seepage runoff and dust fallout from the dump as well as the condition of all water management infrastructure are monitored on an ongoing basis.

20.1.9 Waste Rock Storage Facility

The WRSF is located west of the FRD dam and accommodates all waste rock not used for FRD dam impoundment construction. The WRSF side slopes will be constructed to a gradient of 1:3 and the maximum vertical height of the WRSF will be 25 m.

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

20.1.10 Sites of Archaeological and Cultural Importance

An Archaeological Impact Assessment (AIA) carried out in 2008 revealed several archaeological and burial sites within the KDM and along the access road corridor. 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 most common archaeological occurrences in the Letlhakane area have been materials belonging to as early as the Early Stone Age period (ESA) through to the Iron Age. Middle Stone Age tools have been reported in the Letlhakane Mine area. Overall, this study has indicated that the area was occupied possibly at different times during the Pleistocene. The presence of a small fragment of pottery does point to some use during the past 2000 years.

The burial sites have since 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.

20.1.11 Mine Closure

In terms of Section 65 of the Botswana Mines and Minerals Act (1999), the mine is obliged to develop and implement a mine closure and rehabilitation plan (MCRP) during the life of mine and to ensure that the mining lease area is progressively rehabilitated and ultimately reclaimed at the end of life of mine to the satisfaction of the Director of Mines.

A conceptual mine closure plan for Karowe was incorporated into the pre-mining EIA (approved in 2008) and into the EMP submitted and approved in 2010 following Lucara's takeover of the then AK6 Diamond Mine Project. A conceptual mine closure estimate was developed at the time of commissioning. KDM commissioned Geoflux to develop a detailed MCRP based upon site survey information in 2013 (Geoflux 2013).

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 due to insufficient information on future groundwater impacts and potential treatment costs. Based on the experience of other Botswana diamond mines, it is unlikely that material mine decant will occur during the closure process.

The 2013 MCRP was comprehensively updated in 2018 (DWA 2018), taking into account all potential liabilities associated with the existing operation as well as the UG Project as understood at the time. The current closure plan considers all closure liabilities up to December 2018. As a result of this, a financial guarantee was raised by KDM in August 2019 with respect to the closure liability totalling US$19M. As the mining operation and Botswana mine closure guidance evolves, the closure liability estimates will require further refinement.

Concurrent rehabilitation occurs at the exploration sites, but rehabilitation at Karowe is not scheduled to commence before 2022.

Based on the local climatic and soil conditions, sustainable grazing has been identified as the most appropriate post-closure land-use option and this planning forms part of ongoing consultation with stakeholders to ensure buy-in.

20.2 Socio-Economic Setting

20.2.1 Land Use

KDM is located in the Central District of Botswana, 15 km south-west of the town of Letlhakane to which it is connected via hardened surface road. Letlhakane is a regional centre in central Botswana with a number of diamond mines operating within 75 km to the west and northwest of it.

According to the Central District Integrated Land Use Plan (CDILUP) (Geoflux, 2007), the primary use for tribal land in the sub-district is grazing. The Orapa-Letlhakane region has mixed secondary uses which include arable, settlement and mining activities. The area between Letlhakane and KDM is used for arable and grazing purposes; with grazing becoming more dominant from KDM towards the south, southwest and west. The grazing areas are mainly communal; however, commercial ranches have been demarcated further to the southwest. These ranches, though intended to improve the use and management of land resources, reduce the land available to communal farmers.

The Boteti sub-District has a total of 24 primary schools, six secondary schools and one senior secondary school situated in Letlhakane. There are two junior secondary schools and one senior secondary school in Letlhakane. The Boteti area has an 18.2% HIV prevalence with females being the majority.

The Boteti sub-district has several sanitation infrastructure systems such as the dumping site and the sewage treatment ponds. Waste from the village is taken to the dump site in Letlhakane. The sewage treatment ponds for Letlhakane are operated by water utilities and are functional. KDM has its own waste facilities, including a landfill site, sewage treatment plant and incinerator for which permitting applications have been submitted (see Section 18).

20.3 Social Impact Assessment

The approved EIA (Geoflux, 2007) included a Social Impact Assessment and dedicated stakeholder engagement line functions in the EHS & CR Department to manage stakeholder engagement, social aspects and obligations. Since the project commissioning, the community relations team has been engaging with local stakeholders on an ongoing basis.

As part of the Karowe UG FS, the social impact of the mine and the project were separately assessed and compiled into a separate Social Impact Assessment (SIA) document which maps.

- The 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 underground operation; and
- Current and planned mitigation measures to avoid or ameliorate negative impacts and enhance positive ones.

The findings of this and previous social impact studies show that economic opportunities associated with the mine's operations and expansion, as well as its eventual closure are the primary concern for the majority of stakeholders. To continue to strengthen the engagement process, a Stakeholder Engagement Plan (SEP) was completed in late 2019 which meets the guidance of the International Finance Corporation (IFC) Performance Standards and will guide the engagement activities of the relevant staff going forward.

20.4 Mine Closure

In terms of Section 65 of the Botswana Mines and Minerals Act (1999), the mine is obliged, to develop and implement a mine closure and rehabilitation plan (MCRP) during the Life of Mine and to ensure that the mining lease area is progressively rehabilitated and ultimately reclaimed at the end of life of mine to the satisfaction of the Director of Mines.

A conceptual mine closure plan for Karowe was incorporated into the pre-mining EIA (approved 2008) and the EMP submitted and approved in 2010 following Lucara's takeover of the then AK6 Diamond Mine project. A conceptual mine closure estimate was developed at the time of commissioning, KDM commissioned Geoflux to develop a detailed Mine Closure and Rehabilitation Plan (MCRP) based upon site survey information in 2013 (Geoflux 2013)

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 due to insufficient information on future groundwater impacts and potential treatment costs. Based on the experience of other Botswana diamond mines, it is unlikely that material mine decant will occur during the closure process.

The 2013 MCRP was comprehensively updated in 2018 (DWA 2018), taking into account all potential liabilities associated with the existing operation as well as the UG FS. The current closure plan considers all closure liabilities up to December 2018. As a result of this a Financial Guarantee was raised by KDM in respect of the closure liability for BWP 200 million in August 2019. As the mining operation and Botswana mine closure guidance evolve, the closure liability estimates will require further refinement.

This MCRP sets out site closure options, objectives and criteria for unscheduled closure, and scheduled closure with concurrent rehabilitation, and scheduled closure without concurrent rehabilitation calculating liabilities as set out in Table 20-1.

Table 20-1: Closure Scenario Cost Estimates

Closure Scenario	LOM (M$)
Unscheduled Closure	16.1
Scheduled Closure w/ Concurrent Rehabilitation	16.0
Scheduled Closure without Concurrent Rehabilitation	25.9

Note: Closure estimates were converted from BWP to US$ using an exchange rate of 10.6.
Source: DWA (2018)

Based on the local climatic and soil conditions, sustainable grazing has been identified as the most appropriate post-closure land-use option and this planning forms part of on-going consultation with stakeholders to ensure buy-in.

20.5 Permitting

A list of permits held or in the process of being acquired by the Karowe Diamond Mine is presented in Table 20-2.

Table 20-2: Karowe Diamond Mine Permits

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
EIA Permit	DEA/BOD/CEN/EXT/MNE 015(7)	EIA valid. EMP updated in June 2016 and will be reviewed to include phase 3 in 2018	Dept. of Environmental Affairs	EIA Act
Water Rights	B6615, B6622, B5386, B 5387, B5388, B5389, B7933B7934, B7935, B7936, B7937, B7937, B7938, B7940, B7941, B7942	Valid for the duration of the mining licence	Dept. of Water Affairs	Water Act
Waste Carriers License	CRLIC/649/06-2080/19 - 002 Kellinicks	20/06/2020	Dept. of Waste Management and Pollution Control	Waste Management Act
	CRLIC/649/06-2080/19 - 003 Kellinicks	20/06/2020		
	CRLIC/01/12-063/18- SKIP HIRE	31/12/2019		
Incinerator Permit	Awaiting certificate from the Department of Waste Management and pollution control	Awaiting department of waste management and pollution control to register and licensing the incinerator	Dept. of Waste Management and Pollution Control	Waste Management Act
Borehole Certificates	In Place	Valid for the duration of the mining licence	Dept. of Water Affairs	Boreholes Act
Dumps Classification	All classified	All dumps active	Dept. of Mines	Mines, Quarries, Works and Machinery Act

Statutory Permit	Reference Number	Expiry Date	Responsible Authority	Regulatory Instrument
Surface Rights	LT/SLB/B/1 IV (231)	09/10/2023	Ngwato Land Board	Tribal Land Act
Radiation License	BW0315/2019	Renewed and certificates will expire in 06 November 2021	Radiation Inspectorate	Radiation Protection Act
Waste Facilities & Sewage Plant	Application in Progress	The mine is working on two projects both at the landfill and Sewage plant to address the findings of the Department of Waste Management and Pollution Control	Dept. of Waste Management and Pollution Control	Waste Management Act
License to manufacture explosives	In Place	31/12/2019	Dept. of Mines	Explosives Act
Permit to carry bulk explosives	F35/13, F34/13 and F36/13	31/12/2019	Dept. of Mines	Explosives Act
Magazine License	386:00002948A and 385:00002947A	31/12/2019	Dept. of Mines	Explosives Act
Blasting License for magazine master	In Place	Valid and appointment renewed yearly	Dept. of Mines	Explosives Act

Source: Lucara (2019)

Upon approval of FS in late 2019, the mining lease extension will be completed and filed for the open pit and underground mines with the Government of Botswana (GoB). The underground mine is not expected to require an EIA according to communication with the GoB but an EIA will be needed for the new electrical transmission line.

A new Environmental Management Plan (EMP) will be submitted at the completion of the FS. The new EMP will include reference to:

- The new electrical transmission line
- An updated Closure Plan

A new Stakeholder Engagement Plan is underway andnearing completion with consultation and formal process to begin with approval of FS, preliminary engagement and discussions have taken place.

21 Capital Cost Estimate

21.1 Capital Cost Summary

The capital cost estimate was prepared using a combination of first principles, applying project experience and using vendor / contractor provided budgetary quotes while avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed. The target accuracy of the estimate is ±15%.

Costs are expressed in US$ with no escalation unless stated otherwise. Foreign exchange rates of BWP10.60:US$1.00 and ZAR14.00:US$1.00 are used where applicable.

The estimate is based on the assumption that contractors would mobilize only once to carry out their work and are not already mobilized on site performing other work.

Total life of mine capital costs are estimated to be US$722 M, these include costs to develop the underground as well as current and future sustaining costs for the existing site and open pit operations.

Pre-production capital costs specifically associated with developing the underground amount to $514 M. Capital costs during production years total $208 M. These costs are summarized in Table 21-1. Contingency for the project totals $69 M, with $51 M associated with the underground pre-production capital, and $18 M associated with the LOM sustaining capital costs. Individual contingency rates were applied to each of the capital cost categories by WBS and activity, with rates ranging from 5 to 12.5%. This resulted in a blended contingency rate of 11.2% on the underground pre-production capital, and 10.6% on the overall LOM capital. Closure costs amount to $34 M and were assumed to occur in the two years immediately after plant closure.

Table 21-1: Capital Cost Summary

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
1000 – Mining	321.7	38.1	359.8
2000 – Bulk Earthworks	18.8	-	18.8
3000 – Process Plant	0.1	87.9	88.0
4000 – Fine and Coarse Residue Deposition	-	30.7	30.7
5000 – Onsite Infrastructure	5.9	-	5.9
6000 – Buildings & Facilities	1.6	-	1.6
7000 – Offsite Infrastructure	19.6	-	19.6
8000 – Project Indirects	47.7	-	47.7
9000 – Owner's Costs	46.9	34.0	80.9
Subtotal	**462.1**	**190.7**	**652.9**
Contingency	51.4	17.8	69.2
Total Capital Costs	**513.7**	**208.5**	**722.2**

*numbers may not add due to rounding
Source: JDS (2019)

21.2 Basis of Estimate

The Project pre-production capital estimate includes all costs to develop the UGP to a commercially operable status. The sustaining capital estimate includes all costs to sustain the existing operating site (open pit) and sustain the UGP and the extended operation. Sunk costs and owner's reserve accounts are not considered in the FS estimates or economic cash flows.

The following key assumptions were made during development of the capital estimate:

- The capital estimate is based on the contracting strategy, execution strategy, and key dates described within the Project Execution Plan (PEP) described in Section 25.1 of this report;
- Underground mine development activities will be performed by a contractor until the start of the UG production period (2025); and
- All surface construction (including earthworks) will be performed by local contractors.

The following key parameters apply to the capital estimate:

- Estimate Class: The capital cost estimate is considered a Class 3 feasibility cost estimates (-15%/+15%). The overall project definition is estimated at 30%;
- Estimate Base Date: The base date of the capital estimate is Q3 2019. No escalation has been applied to the capital estimate for costs occurring in the future. Proposals and quotations supporting the FS estimate were received in Q2 and Q3 of 2019;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate, except pipe sizes which are included in Nominal Pipe Size (NPS) inches; and

- Currency: All capital costs are expressed in US$. Table 21-2 presents the exchange rates used for costs estimated in foreign currencies.

Table 21-2: Foreign Currency Exchange Rates

US$	Exchange Rates	Currency
1 US$ =	1.33	C$
	10.6	BWP
	14.0	ZAR

Source: JDS (2019)

21.3 Mine Capital Cost Estimate

21.3.1 Open Pit Mining

The open pit operation is fully contracted, and as such there are minimal capital costs associated with the five-year mine plan budget. Any minor planned capital costs have been included within the sustaining costs included within the sustaining budget of the current operation categorized under WBS 3000.

21.3.2 Underground Mining

Underground capital costs contain a mix of first principal, vendor supplied, and database quotes using local suppliers and currencies where possible. Time related costs for development or infrastructure installations have been estimated by JDS or by third party vendors and contractors.

Underground capital costs are summarized in Table 21-3.

Table 21-3: Underground Capital Costs

Capital Costs	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Surface Infrastructure	2.1	2.1	0.0	1%
Underground Equipment	57.7	27.2	30.5	16%
Underground Infrastructure	11.0	10.9	0.1	3%
Underground Development	70.7	70.7	0.0	20%
Underground Systems	38.7	35.3	3.4	11%
Capitalized UG Operating Costs	15.4	15.4	0.0	4%
Shaft Sinking and Infrastructure	164.2	160.1	4.2	46%
Total Mining	**359.8**	**321.7**	**38.1**	**100%**

Source: JDS (2019)

21.3.3 Labour

Underground mining staffing levels are built up based on the productivities (man-hours) required for capital development and installation activities occurring within a given time period. As such, mining manpower fluctuates throughout the capital development period.

The labour workforce responsible for capital development will be almost entirely contracted, less existing on-site owner's team management. Contractor labour rates are based on the existing open pit contractor rates plus a 25% mark-up for an underground allowance, a 35% mark-up for any contractors assumed to be expatriate, and a 15% mark-up for profit and tooling. The mine plan envisions three primary contractors working to develop the mine with support of four additional sub-contractors to manage specific procurement packages. These contractors are summarized as:

Primary Contractors

- Shaft Sinking Contractor:
    - Responsible for all shaft sinking, equipping, and commissioning.
- Underground Development Contractor:
    - Responsible for all lateral underground development, long hole raise development, and pre-production stope production.
- Raise bore Contractor:
    - Responsible for all underground raise bore development

Sub-contractors

- Concrete Contractor:
    - Supply and installation of concrete on surface and underground. Includes preparation of foundations, pedestals, columns, and structural walls as required.
- Structural Steel Contractor:
    - Supply and installation of structural steel on surface and underground (excluding shaft requirements).
- Mechanical Contractor:
    - Assembly and installation of equipment on surface and underground, including compressors, pumps, fans, coolers, shops, refuge chambers, doors, and crushing & conveying equipment.
- Electrical Contractor:
    - Installation of electrical equipment and cabling not performed by the underground development contractor.

Several existing contracts supporting the open pit operations will continue to support the underground operations, including the batch plant and explosives production facility.

Annual contractor salaries are based on working two 12-hour shifts per day and account for all overtime and burdens. Burdens amount to 51% of the base salary, and account for items such as housing, gratuity, medical, vacation, group insurance, and for those eligible, cell phone and car allowance.

A summary of the primary contractor labour requirements are located in Table 21-4 to Table 21-6. Shaft and raise bore labour requirements have been provided by contractor estimates. Lateral development labour requirements have been estimated from first principals using the same logic as applied to the operating cost labour.

Table 21-4: Shaft Contractor Labour Requirements

Shaft Contractor	Units	Roster	Peak	Average
Administration				
Total Employed	#	4x2	8	7
Average Day Shift	#	4x2	4	3
Average Night Shift	#	4x2	2	2
Supervision				
Total Employed	#	4x2	25	15
Average Day Shift	#	4x2	21	12
Average Night Shift	#	4x2	2	2
Surface Crew				
Total Employed	#	4x2	52	27
Average Day Shift	#	4x2	26	13
Average Night Shift	#	4x2	11	6
Sinking Crew				
Total Employed	#	4x2	58	34
Average Day Shift	#	4x2	19	11
Average Night Shift	#	4x2	19	11
SHEQT*				
Total Employed	#	4x2	11	9
Average Day Shift	#	4x2	9	7
Average Night Shift	#	4x2	0	0

*Safety, Health, Environmental, Quality and Training
Source: JDS (2019)

Table 21-5: Development Contractor Labour Requirements

Development Contractor	Units	Roster	Peak	Average
Administration				
Total Employed	#	4x2	29	23
Average Day Shift	#	4x2	14	11

Development Contractor	Units	Roster	Peak	Average
Average Night Shift	#	4x2	4	3
Development Crews				
Total Employed	#	4x2	130	95
Average Day Shift	#	4x2	35	26
Average Night Shift	#	4x2	30	22
Maintenance				
Total Employed	#	4x2	51	37
Average Day Shift	#	4x2	14	10
Average Night Shift	#	4x2	11	8

Source: JDS (2019)

Table 21-6: Raise Bore Contractor Labour Requirements

Raise Bore Contractor	Units	Roster	Peak	Average
Administration				
Total Employed	#	4x2	7	5
Average Day Shift	#	4x2	4	3
Average Night Shift	#	4x2	1	1
Development Crew				
Total Employed	#	4x2	17	13
Average Day Shift	#	4x2	5	4
Average Night Shift	#	4x2	5	4

Source: Master Drilling (2019)

21.3.4 Surface Infrastructure

Surface infrastructure capital costs are summarized in Table 21-7. Other surface infrastructure costs are included in the general site infrastructure estimates.

Table 21-7: Mine Capital - Surface Infrastructure

Surface Infrastructure	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Power Supply and Distribution (at shafts)	1.2	1.2	0.0	60%
Buildings	0.8	0.8	0.0	40%
Total	**2.1**	**2.1**	**0.0**	**100%**

Source: JDS (2019)

The costs include the following:

- Electrical distribution from the surface substation to the headframes, hoist house, and compressor building; and
- Compressor building and components.

21.3.5 Underground Mobile Equipment

Underground mobile equipment capital costs are summarized in Table 21-8 and exclude shaft equipment.

Table 21-8: Mine Capital - Underground Equipment

Underground Equipment	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Drilling	8.6	8.6	0.0	15%
Charging	1.0	1.0	0.0	2%
Loading	10.0	8.6	1.4	17%
Hauling	4.7	3.5	1.2	8%
Ground Support	4.0	3.0	1.0	7%
Services	0.3	0.3	0.0	1%
Ancillary	2.2	2.2	0.0	4%
Equipment Overhauls	13.2	0.0	13.2	23%
Equipment Replacements	13.8	0.0	13.8	24%
Total	**57.7**	**27.2**	**30.5**	**100%**

Source: JDS (2019)

Underground mining equipment quantities and costs were determined through build-up of mine plan quantities and associated equipment utilization requirements. Quotes were received from local vendors and applied to the required quantities.

Vendor quotes include both a base price plus the price of applicable accessories to represent the loaded price. The loaded price includes on site assembly and commissioning of the equipment by a vendor representative. Capital spares at 10% of the loaded price account for major components including drivetrains, motors, and rock drills. Freight costs have been provided by the vendor from the nearest port to the project site. Where freight costs were not provided, a 5% freight charge has been applied to the loaded price. First fills at 0.5% of the loaded price account for initial fuel, lubrication, and supply of consumables including ground engaging equipment. Cost to disassemble and reassemble (if required) and sling equipment underground to the working location has been included in the capital cost of equipment. This cost is calculated from first principals where possible, and otherwise applied as a nominal 1.5% of the loaded price.

Mobile equipment costs have been scheduled in two ways. Initially all mobile equipment will be supplied by the primary development contractors with the cost of ownership charged to KDM on an hourly or monthly rate. The contractor will charge a 15% mark-up on all equipment consumables (excluding fuel), as well as a monthly ownership charge equal to 4% of the loaded price assuming a maximum 450 operating hours per month. These costs are incurred as part of the lateral development cost and not captured under equipment capital.

During commercial operation KDM will self-perform all mining operations and will purchase the required mobile equipment fleet. This equipment will be purchased with 20% down payment 12 months in advance of requirement, with the remaining 80% paid upon delivery. Equipment will be brought on site three months in advance of being required underground. All mobile equipment purchases are assumed new, and no provision for purchasing contractor equipment has been accounted for. These options are to be reviewed closer to the time of commercial operations.

All mobile equipment will come supplied with roll-over protection, fire protection, and the latest on-board safety technology including tramming cameras and alarms, proximity detection systems, and emergency steering. Where possible, equipment will be outfitted with enclosed cabins and air conditioning to protect against heat stress. Drills will be outfitted with onboard air compressors for flushing holes and drilling systems to pre-program and automate drill patterns. Auto-lubrication and foam filled tires will be applied where possible to reduce wear on equipment.

The production LHDs will be operated in a manual arrangement. Equipment automation tooling, whereby operators sit in a control room away from the equipment, has not been included in the capital estimate.

A mid-life major overhaul is budgeted for all equipment equal to 60% of the base price of the unit. Equipment will be replaced with new units at the end of the expected equipment life. Equipment will not be replaced within one year of mine closure and will instead be operated at a higher cost of maintenance.

Table 21-9 lists the LOM equipment purchases, rebuilds, and replacements.

Table 21-9: Mine Equipment Capital Costs

Equipment	Unit Cost ($M)	LOM Purchases	Rebuild Frequency (x1000 hours)	LOM Rebuilds	Replacement Frequency (x 1000 hours)	LOM Replacements
LHD (17t/7.0m^3)	1.6	2	12.5	0	25.0	0
LHD (21t/8m^3)	1.7	3	12.5	6	25.0	3
FEL (15t/5.4m^3)	1.6	1	24.5	0	49.0	0
Surface Truck	1.2	4	35.0	4	70.0	0
Jumbo - 2 Boom	1.6	1	9.0	0	18.0	0
Longhole Drill - ITH	1.4	5	7.5	5	15.0	2
Secondary Breakage Drill	1.1	2	10.0	2	20.0	4
Bolter	1.2	2	12.5	0	25.0	0
Cable Bolter	1.5	1	7.5	0	15.0	0
Shotcrete Sprayer	0.0	1	5.0	0	10.0	0
Small Explosives Truck	0.3	1	10.0	0	20.0	0
Large Explosives Truck	0.3	2	10.0	0	20.0	0
Transmixer	0.3	1	10.0	2	20.0	0
Scissor Lift	0.2	1	10.0	0	20.0	0

Equipment	Unit Cost ($M)	LOM Purchases	Rebuild Frequency (x1000 hours)	LOM Rebuilds	Replacement Frequency (x 1000 hours)	LOM Replacements
Fuel/Lube Truck	0.1	1	14.0	2	28.0	1
Mechanics Truck	0.1	1	14.0	1	28.0	1
Electrician Truck	0.1	1	14.0	1	28.0	0
Boom Truck	0.3	1	10.0	1	20.0	1
Grader	0.3	1	10.0	1	20.0	0
Telehandler	0.2	1	10.0	2	20.0	1
Supervisor Truck	0.1	6	14.0	8	28.0	0
Utility Vehicle	0.1	6	14.0	0	28.0	0
Ambulance	0.1	1	14.0	0	28.0	0

Source: JDS (2019)

21.3.6 Underground Infrastructure

Underground infrastructure capital costs are summarized in Table 21-10.

Table 21-10: Mine Capital - Underground Infrastructure

Underground Infrastructure	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Crusher and Conveyor	4.4	4.4	0.0	40%
Maintenance Shop & Services	3.0	3.0	0.0	28%
Sumps and Pumping Facilities	1.4	1.4	0.0	12%
Doors and Bulkheads	2.0	1.9	0.1	18%
Primary Refuge and Lunchroom	0.2	0.2	0.0	1%
Total	**11.0**	**10.9**	**0.1**	**100%**

Source: JDS (2019)

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling. Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items. Acquisition of underground infrastructure is timed to support the mine plan requirements.

The crusher and conveyor costs include the installation and commissioning of the structural steel, concrete, and mechanical components of the system. Costs associated with electrical installations and chamber excavation are carried elsewhere.

The maintenance shop and services include a multi-bay workspace to perform maintenance and repair, refueling, lubrication and washing, as well as store parts and consumables. The cost of excavation and ground support has been captured under the lateral development capital costs. Maintenance facility capital costs include the supply and install of all floor preparations, overhead cranes, fuel stations, fire suppression, and tooling.

Sumps and pumping facilities include the supply and install of all furnishings including concrete, piping, catwalks, chain hoists, and beam trollies, excluding pumps.

Doors and bulkheads include the supply and install of man doors, fire doors, air locks, fan bulkheads, and regulators. Time and material for blocking around the doors are included.

The primary refuge and lunchroom is a dual-purpose area which will serve as a daily lunchroom as well as emergency refuge chamber. The lunchroom will include items such as a latrine, washing facilities, concrete floor, concrete blocked man doors, safety equipment, fire suppression, lighting, and benches.

21.3.7 Underground Development

Underground development capital costs are summarized in Table 21-11.

Table 21-11: Mine Capital - Underground Development

Underground Development	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
680 Development	4.7	4.7	0.0	7%
580 Development	14.9	14.9	0.0	21%
480 Development	8.1	8.1	0.0	11%
380 Development	4.2	4.2	0.0	6%
310 Development	29.1	29.1	0.0	41%
Raises	6.8	6.8	0.0	10%
Development Contractor Mob/Demob	2.9	2.9	0.0	4%
Total	**70.7**	**70.7**	**0.0**	**100%**

Source: JDS (2019)

Underground development includes all work completed by the development and raise bore contractor and does not include shaft sinking. Capital costs include the mobilization and demobilization of both contractors. Cost for raise bore contractor mobilization was provided by a raise bore contractor budgetary quote. Cost for development contractor mobilization was estimated based on the following criteria:

- $250,000 allowance for temporary facilities;
- $2,000 per contractor to account for transport, induction training, and PPE;
- Freight costs of mobile equipment; and
- $100,000 allowance for first fills.

Development costs account for the labour, equipment, materials, fuel, and supervision required to drive all lateral and vertical development prior to commercial production. Development furnishings include ground support, ventilation bagging, communication and power cables, and piping for air and water.

Lateral and vertical development unit costs are summarized below.

- Lateral development blended unit cost - $4,363/m:
    - Blended rate of 5.0 x 5.0 m (58%), 5.5 x 5.5 m (26%), 6.0 x 6.0 m (12%), 8.0 x 6.5 m (1%), and 2.0 x 2.0 m (3%).
- Raise bore development:
    - 3.0 m diameter - $4,730/m; and
    - 4.0 m diameter - $4,960/m.
- Drop raise (3.0 x 3.0 m) – 1,627/m.

21.3.8 Underground Systems

Underground systems capital costs are summarized in Table 21-12.

Table 21-12: Mine Capital - Underground Systems

Underground Systems	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Electrical Distribution	13.8	13.8	0.0	36%
Ventilation Equipment	2.4	2.2	0.2	6%
Mine Cooling Equipment	8.3	7.8	0.5	21%
Pumping Equipment	9.2	7.9	1.3	24%
Underground Communications	1.1	0.5	0.6	3%
Portable Refuge Chambers	0.6	0.3	0.3	1%
Mine Safety	3.3	2.7	0.6	8%
Total	**38.7**	**35.3**	**3.4**	**100%**

Source: JDS (2019)

Electrical distribution costs include the supply and install of all shaft and level cabling, junction boxes, substations, and mine power centers.

Ventilation equipment costs include the supply and install of all permanent fans, mounting equipment, and start-up supply of rigid and bagged ducting. An annual sustaining capital cost equal to 4% of the fan purchases has been included to account for maintenance and replacements over time.

Mine cooling equipment costs include the supply and install of all chillers, cooling cars, distribution piping, heat rejection chambers, and water storage containers.

Pumping equipment costs include the supply and install of all pumps for sumps, booster stations, and development faces. An annual sustaining capital cost equal to 6% of the pump purchases has been included to account for maintenance and replacements over time.

The UG communications cost includes the supply and install of the wireless communication system, leaky feeder system, handheld radios, signage, and barriers.

Portable refuge chamber costs include the supply and install of 20-man refuge chambers.

Mine safety costs include the supply and install of mine rescue equipment, cap lamps, hand-held gas monitors, a stench gas system, cavity monitoring systems, geotechnical monitoring equipment, and initial provision of technical tooling (mine software, survey equipment, pull testing gear). Pre-production and sustaining costs for PPE are included, as well as 20 hours a month for consultant services.

21.3.9 Capitalized Operating Costs

Capitalized operating costs are summarized in Table 21-13.

Table 21-13: Mine Capital – Capitalized Operating Costs

Capitalized Operating Costs	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Production Stoping	1.8	1.8	0.0	12%
Crushing & Hoisting	2.7	2.7	0.0	18%
Mine Maintenance	1.2	1.2	0.0	8%
Mine General	9.7	9.7	0.0	63%
Total	**15.4**	**15.4**	**0.0**	**100%**

Source: JDS (2019)

Capitalized operating costs account for the labour, equipment, materials, fuel, and supervision required to perform all stoping, mucking, crushing, and hoisting of production ore during the pre-production period.

Stoping costs are broken down in Table 21-14. During pre-production, all stoping activities will be performed by the development contractor.

Table 21-14: Mine Capital - Pre-Production Stoping Unit Costs

Operating Costs	Estimated Annual	Pre-Production	Unit Cost per tonne Processed	Weighting
	(M$)	(M$)	US$/t	%
Labour	1.4	0.6	2.12	32%
Equipment	1.6	0.7	2.42	37%
Material	0.7	0.3	1.10	17%
Fuel	0.5	0.2	0.82	13%
Power	0.1	0.0	0.08	1%
Total	**4.4**	**1.8**	**6.54**	**100%**

Source: JDS (2019)

Crushing hoisting costs account for all labour, equipment, fuel, maintenance, and power consumption associated with operating the crusher and conveyor during pre-production.

Mine maintenance includes the labour, material, and tooling required to service the fixed and mobile equipment during pre-production. This work will be performed by the development contractor.

Mine general costs include supervision labour and support equipment, as well as infrastructure power to operate the ventilation and dewatering systems during pre-production.

21.3.10 Shaft Sinking and Infrastructure

Capital shaft sinking and infrastructure costs are summarized in Table 21-15.

Table 21-15: Mine Capital – Shaft Sinking and Infrastructure

Shaft Sinking & Infrastructure	LOM Total	Pre-Production	Sustaining	Weighting
	(M$)	(M$)	(M$)	%
Common Preliminaries and Generals	1.4	1.4	0.0	1%
Production Headframe, Hoist, & Pre-Sink	6.4	6.4	0.0	4%
Ventilation Headframe, Hoist, & Pre-Sink	4.5	4.5	0.0	3%
Production Shaft Sinking	75.1	75.1	0.0	46%
Ventilation Shaft Sinking	60.1	60.1	0.0	37%
Shaft Equip & Commission	7.9	7.9	0.0	5%
Shaft Indirect Costs During UG Development	4.5	4.5	0.0	3%
Shaft Sustaining Capital	4.3	0.1	4.2	3%
Total	**164.2**	**160.1**	**4.2**	**100%**

Source: JDS (2019)

Shaft sinking and equipping is the single largest capital cost in the mine. Shaft capital estimates and construction durations were prepared by United Mining Services (UMS) and scheduled by JDS. Costs for the shaft include the purchase of one of three currently available used headframes. The cost and refurbishment of the used equipment does not offer costs savings but improves the delivery schedule. All other shaft equipment is priced as new.

The all-in unit cost to sink and equip the production shaft is $120,000/m. The average cost to sink and equip the ventilation shaft is $89,000/m.

An annual sustaining capital cost equal to 1% of the shaft mechanical purchases has been included to account for maintenance and replacements over time. Additional preventative maintenance costs have been included in the shaft operating costs.

21.4 Processing Capital Cost Estimate

The processing of ore from underground is not anticipated to have a material change on the overall plant design or operation. A cost for additional metal detection has been included in the pre-production estimate, based on vendor quotes.

Sustaining capital costs in the process WBS include all the current, and future stay in business costs to continue to operate the plant, and site infrastructure outside of the mine. These costs are based on the current Karowe five-year capital budget costs, which is derived from a combination of historical costs, engineered plans and vendor quotes developed by the current site operations team. The five-year capital budget costs have then been extrapolated over the remaining LOM.

Table 21-16: Process Costs

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
3000 – Process Plant	0.1	87.9	88.0

Source: JDS (2019)

21.5 Infrastructure Capital Cost Estimate

Surface construction costs include site development, fine residue deposition facility, and on-site and off-site infrastructure. These cost estimates are primarily based on material and equipment costs from MTO's and detailed equipment lists. Pricing for main equipment and bulk materials was primarily determined from quoted sources, with some factors applied for minor cost elements.

Table 21-17 presents a summary basis of estimate for the various commodity types within the surface construction estimates. Growth factors were included above neat material take-off quantities for all areas.

Table 21-17: Surface Infrastructure Basis

Description	Basis
Pre-engineered Buildings, modular buildings and warehouses.	Buildings sized according to general arrangements, with quotations for overall building structures from local vendors.
Services to Buildings	Estimated based on site provided data for similar projects
Bulk Earthworks and Roads	Material take-offs for surface works and roads from preliminary 3D model. Unit rates from first principles based on local contractor rate sheets
Mechanical Equipment	Vendor quotes or current site-based pricing for similar equipment
Overland Piping	MTO's for major pipelines with supply and installation costs derived from existing pricing from similar current site projects.
Electrical	Major electrical equipment list prepared and detailed major cable runs prepared in neat line material take-offs. Major equipment and cabling based on subcontractor quotes.
Concrete	MTO's measured in neat quantities and quoted rates from local subcontractors

Source: JDS (2019)

A summary of the surface infrastructure costs is outlined in Table 21-18. The current Karowe five-year capital budget includes a provision for the expansion of the FRD facilities to accommodate the material processed from the open pit as part of the existing mine plan. This cost has been included as part of the sustaining costs included existing five-year plan as outlined in Section 21.4. The additional costs to expand the FRD facilities to accommodate the material produced from the UG operation have been included as sustaining costs in Table 21-18.

Table 21-18: Surface Infrastructure Costs

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
2000 – Bulk Earthworks	18.8	-	18.8
4000 – Tailings (CRD and FRD)	-	30.7	30.7
5000 – Onsite Infrastructure	5.9	-	5.9
6000 – Buildings & Facilities	1.6	-	1.6
7000 – Offsite Infrastructure	19.6	-	19.6
Total	**45.9**	**30.7**	**76.6**

Source: JDS (2019)

21.6 Indirect Capital Cost Estimate

Indirect costs are classified as costs not directly accountable to a specific cost object. Table 21-19 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-19: Basis for Indirect Costs

Description	Basis
General Construction Services	Allowances for temporary facilities and support services based on quotes from local vendors and local labour rates with projected requirements based on project scope and schedule.
Construction Camp	Camp sized according to the General Arrangement with contractor quotes for the supply and setting of the facilities. Site utilities based on existing site project information for similar activities. Operations based on first principles and local labour rates, and quotations from local caterers.
Contractor Field Indirects	Estimated from contractor quotes, and including the following items: Time based cost allowance for general construction site services (temporary power, contractor support, etc.) applied against the surface construction schedule Construction offices and wash car facilities Safety training, tools and equipment Environmental cost Materials management and warehouse operations Site maintenance and temporary services Surveying and quality assurance Communications Contractor facilities and related cost
Temporary Power	Temporary power requirements, prior to the commissioning of the expanded BPC line, are based a construction specific electrical load list. Costs include both the supply and maintenance of temporary generators while required, along with the costs of generating power to meet the project demand during construction. Costs are based on site specific requirements and local vendor quotes.

Description	Basis
Flights & Travel	Based on detailed project labour build up and projected travel requirements, with quotes from local / regional service providers
Freight	Where freight has not been included as part of a vendor quote, costs have been developed from equipment weights and quotes from regional vendors.

Source: JDS 2019

21.7 Owner's Cost Estimate

Owner's costs are classified as the management, oversight and site operation costs that are incremental costs to develop the UGP. These costs are capitalized during the construction phase. Any owner's costs that continue beyond the project phase are then incorporated into the site G&A operating costs. Table 21-20 presents the subjects and basis for the owner's costs within the capital estimate.

Table 21-20: Basis for Owner's Cost

Description	Basis
Engineering & Procurement	Detailed man-hour estimate based on deliverables for engineering and drafting, and time based on project management services required to oversee project development. Costs are based on an EPCM execution strategy. A schedule of rates was applied against a staffing plan. Estimates for detailed engineering have been provided by suitable sub-consultants as required.
Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration. Costs are based on an EPCM execution strategy. A schedule of rates was applied against a staffing plan.
Owner's Project Team	Detailed man-hour estimate, based on the incremental requirements identified by Lucara and local labour rates.
Taxes	Value Added Tax (VAT) has been assumed to be recoverable and not included in the Capital estimates. Withholding taxes on out of country consulting labour of 10% (regional) and 15% (international) have been applied to consulting services within the EPCM estimate.
Community Relations	Excluded and not part of FS costs
Escalation	Excluded (but sensitivities to be provided with economic model)

Source: JDS (2019)

21.8 Closure Cost Estimate

The Mine Closure Reclamation Plan (MCRP) sets out site-specific closure options, objectives and criteria for unscheduled closure, scheduled closure with concurrent rehabilitation, and scheduled closure without concurrent rehabilitation. These costs are presented in Table 21-21.

Table 21-21: Closure Cost Estimates

Closure Scenario	LOM (M$)
Unscheduled Closure	16.1
Scheduled Closure w/ Concurrent Rehabilitation	16.0
Scheduled Closure without Concurrent Rehabilitation	25.9

Note: Closure estimates were converted from BWP to US$ using an exchange rate of 10.6.
Source: DWA (2018)

21.9 Contingency

Contingency was applied to the capital costs based on the contingency matrix outlined in Table 21-22. Detail on the mine capital contingencies is provided in Table 21-23. Contingency was determined based on experience on similar projects, the level of detail in engineering design and associated pricing and quotes. Equipment and infrastructure that had firm quotes were given a lower contingency percentage than activities such as shaft sinking, or dewatering drilling that are dependent on productivity through ground conditions that may still be uncertain.

Table 21-22: Contingency

Capital Cost Category	Labour	Perm Equipment	Equip	Other
	(%)	(%)	(%)	(%)
On-Site Development	10	10	10	10
Dewatering	-	-	-	15
Process Plant	10	10	10	10
CRD/FRD & Mine Waste Management	10	10	10	10
On-Site Infrastructure	10	5	10	10
Buildings & Facilities	10	5	10	10
Off-Site Infrastructure	10	10	10	10
Indirect Costs	5	-	-	10
Owner's Costs	10	-	-	10

Source: JDS (2019)

Table 21-23: Mine Cost Contingencies

Capital Cost Category	Labour	Materials	Equipment	Other
	(%)	(%)	(%)	(%)
Mining - Surface infrastructure	10	10	10	10
Underground Equipment	5	5	5	5
Underground Infrastructure	10	10	10	10
Underground Development	12	12	12	12
Underground Systems	10	15	10	10

Capital Cost Category	Labour (%)	Materials (%)	Equipment (%)	Other (%)
Capitalized Underground Production Costs	10	10	10	10
Shaft Sinking and Infrastructure	12.5	12.5	12.5	12.5

Source: JDS (2019)

A higher contingency, 15%, was applied to lateral development planned above 480 masl where ground conditions are anticipated to be worse than the contingency placed on development in lower levels, 10%, where ground conditions are anticipated to be more competent. Table 21-24 outlines the LOM contingencies related to the mine costs.

The LOM initial and sustaining mine capital contingency is 11.1%. Pre-production contingency is 11.7%.

Table 21-24: Underground Mine Capital – Contingency

Contingency	LOM Total (M$)	Pre-Production (M$)	Sustaining (M$)	Weighting %
Surface Infrastructure	0.2	0.2	0.0	1%
Underground Equipment	2.9	1.4	1.5	7%
Underground Infrastructure	1.1	1.1	0.0	3%
Underground Development	8.1	8.1	0.0	20%
Underground Systems	5.7	5.2	0.5	14%
Capitalized UG Operating Costs	1.5	1.5	0.0	4%
Shaft Sinking and Infrastructure	20.5	19.9	0.5	51%
Total	**39.9**	**37.4**	**2.5**	**100%**

Source: JDS (2019)

22 Operating Cost Estimate

22.1 Operating Cost Summary

As the KDM is currently in operations, the operating cost estimates for processing, open pit mining and site G&A were prepared using historical site data and forecast operating budget information provided by Lucara. Adjustments were made to the provided budget information to incorporate adjustments in power costs and changes in labour requirements to support the UG. The UG mining operating costs were prepared using first principles, applying project experience and avoiding the use of general industry factors. Inputs are derived from engineers, contractors and suppliers who have provided similar services to other projects.

Operating costs in this section of the report include mining, processing, coarse and fine residue deposition, and administration up to the production of diamonds from site. Off-site, in-country corporate costs such as Lucara Botswana management, cost of sales, and costs associated with Clara have been provided by Lucara and are included as sales and corporate costs in the economic model. UG mine operating costs incurred during the construction phase are capitalized and form part of the capital cost estimate. All other operating costs incurred during the construction phase to support the current operations are included as part of operating costs.

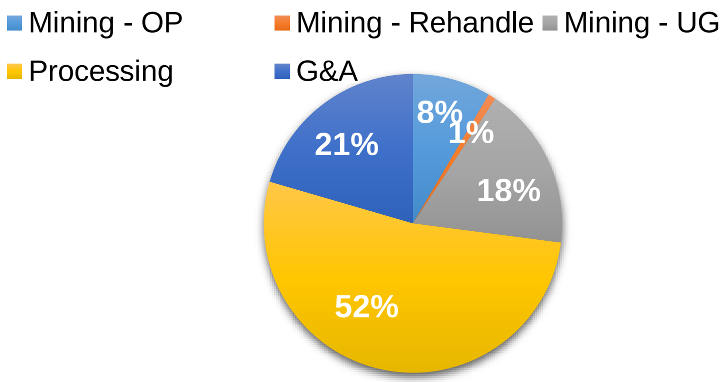
Operating costs are presented in 2019 US dollars on a calendar year basis. No escalation or inflation is included. Total on-site operating costs over the life of mine are $1,593 M and are summarized in Table 22-1. Figure 22-1 shows the breakdown and distribution of the LOM operating costs by category.

Table 22-1: Breakdown of Estimated Operating Costs

Operating Costs	US$/t processed	LOM M$
Mining	7.77	435.4
Processing	14.88	833.4
G&A	5.77	323.2
On-site Total	**28.42**	**1,592.6**
Sales and Corporate Costs	4.58	256.5
Total Operating Costs	**33.00**	**1,849.1**

Source: JDS (2019)

Figure 22-1: Breakdown of Estimated Operating Costs



Source: JDS (2019)

Operational labour rates have been estimated by applying legal and discretionary burdens against base labour rates. Wage scales were defined and applied to the various operational positions based on skill level and expected salary based on the Patterson Job Grading Methodology, consistent with current operational practice. Lucara Botswana human resources personnel were involved in the buildup and verification of the operational labour rates.

Main operating costs component assumptions are shown in Table 22-2.

Table 22-2: Main Cost Assumptions

Item	Unit	Value
Electrical Power Cost (line power)	$/kWh	0.0897
Diesel Cost (delivered)	$/litre	0.816

Source: JDS (2019)

22.2 Mine Operating Cost Estimate

22.2.1 Open Pit Operating Costs

KDM currently operates an open pit mine. Open pit mine operating costs are based on past performance, current budgets, and account for any forecasted adjustments to the open pit operating strategies.

Open pit operating costs are based on the five-year budget prepared by Lucara in September 2019. Open pit operations are currently performed by a mining contractor. The existing contract mining rates for mine operations and rehandling activities were used to update the five-year budget based on the combined open pit and underground mine production schedule. Incremental costs for mining at depth and haulage to WRSF destinations were applied according to the existing contract. The existing contract mining rates and the five-year budget were used to forecast open pit operating costs beyond 2024. These costs are listed in Table 22-3.

Table 22-3: Open Pit Mining Operating Cost Summary by Activity

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Contractor Mining Operations	22.7	136.2	8.25	-
Contractor Mining Rehandle	0.6	13.6	1.16	-

Source: JDS (2019)

22.2.2 Underground Operating Costs

Underground operating costs refer to expenses incurred after the start of underground commercial production and includes all activities directly related to the drilling, blasting, loading, and hauling of ore to the processing facility and waste to the storage facility.

The UG mining operating costs include the following functional areas:

- Development - costs related to the drilling, blasting, mucking, and hauling of development ore and waste. It should be noted that 87% of UG development is captured in capital costs so there is very little operating development;
- Production - costs related to the ITH drilling, blasting, and mucking of ore;
- Crushing & Hoisting – costs related to the operation and maintenance of the underground crusher, conveyor, and shaft equipment, as well as surface haulage equipment;
- Mine Maintenance – costs related to the maintenance of underground fixed and mobile equipment;
- Mine General - costs related to mine support activities such as supervision, technical services, shared infrastructure, support equipment, and material delivery underground; and
- Contingency – a 5% nominal cost applied to all areas of mine operating costs.

Table 22-4: Underground Mining Operating Cost Summary by Activity

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Mined	Weighting
	M$	M$	$/t	%
Development	0.6	7.4	0.22	3%
Production	7.3	94.9	2.90	33%
Crushing & Hoisting	4.8	62.7	1.91	22%
Mine Maintenance	2.7	35.6	1.09	12%
Mine General	5.5	71.3	2.18	25%
Contingency	1.0	13.6	0.42	5%
Total UG Mining OPEX	**21.9**	**285.4**	**8.72**	**100%**

Source: JDS (2019)

Table 22-5: Mining Operating Cost Summary by Area (excluding mine G&A)

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Labour	8.4	109.6	3.35	38%
Equipment	4.4	57.5	1.76	20%
Material	3.1	40.2	1.23	14%
Fuel	1.6	20.3	0.62	7%
Power	3.4	44.4	1.36	16%
Contingency	1.0	13.6	0.42	5%
Total UG Mining OPEX	**21.9**	**285.4**	**8.72**	**100%**

Source: JDS (2019)

22.2.2.1 Underground Operating Cost Stages

There are two distinct operating cost stages during the KDM underground operations:

Stage 1 – Drill and Blast

In this stage the South Lobe is drilled and blasted to “shrink” the reserves. All 33 Mt of ore will be drilled and blasted in the first six years of operations while LHDs muck out the swell at a constant rate. This stage experiences higher than average operating costs.

Stage 2 – Draw Down

In this stage the South Lobe has been fully blasted and the LHDs continue to muck at the same rate as Stage 1 until all of the underground reserves have been drawn from the South Lobe. This stage experiences a lower than average operating cost.

A summary of mining operating costs by stage is located in Table 22-6.

Table 22-6: Underground Mine Operating Cost by Stage

Operating Costs	Stage 1 Drill & Blast	Stage 2 Draw Down	LOM Average
	$/t	$/t	$/t
Lateral Development	0.48	0.00	0.22
Production Stoping	4.15	1.79	2.90
Crushing & Hoisting	1.89	1.94	1.91
Mine Maintenance	1.23	0.95	1.09
Mine General	2.30	2.08	2.18
Contingency	0.50	0.34	0.42
Total UG Mining OPEX	**10.54**	**7.09**	**8.72**

Source: JDS (2019)

22.2.2.2 Underground Mining Labour

Underground mining staffing levels are built up based on the productivities (man-hours) required for mining activities occurring within a given time period. As such, mining manpower fluctuates throughout the mine life.

Underground labour rates are based on the existing open pit labour force plus a 25% mark-up for an underground allowance. Rates include all overtime and burdens associated with 12-hour shifts. Burdens amount to 51% of the base salary, and account for items such as housing, gratuity, medical, vacation, group insurance, and for those eligible, cell phone and car allowance. Expatriate labour rates have been benchmarked against publicly available underground miner salaries within South Africa to ensure that KDM will be able to attract the talent required for specialty positions.

Table 22-7: Underground Labour Cost Summary

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Lateral Development	0.2	2.5	0.08	2%
Production Stoping	1.7	22.1	0.67	20%
Crushing & Hoisting	2.0	25.4	0.78	23%
Mine Maintenance	2.5	32.1	0.98	29%
Mine General	2.1	27.4	0.84	25%
Total Mining OPEX	**8.4**	**109.6**	**3.35**	**100%**

Source: JDS (2019)

Note that underground labour costs do not include mine management and technical roles that are already employed by KDM and captured under general and administrative costs. Only those management and technical roles required in addition to the existing workforce is captured within the mine operating costs. The value of the mining workforce captured within general and administrative costs equates to approximately $1.15 per tonne ore processed.

A summary of labour positions by category during Stage 1 and Stage 2 of the mine plan is provided in Table 22-8.

Table 22-8: Underground Mine Operating Labour Requirements

Operating Cost Labour	Units	Labour Type	Roster	Stage 1 Drill & Blast	Stage 2 Draw Down
Mine General					
Total Employed	#	Staff	5x2	18	18
Average Day Shift	#	Staff	5x2	10	10
Average Night Shift	#	Staff	5x2	3	3
Technical Services					
Total Employed	#	Staff	5x2	9	9

Operating Cost Labour	Units	Labour Type	Roster	Stage 1 Drill & Blast	Stage 2 Draw Down
Average Day Shift	#	Staff	5x2	7	7
Average Night Shift	#	Staff	5x2	0	0
Lateral Development					
Total Employed	#	Hourly	4x4	52	0
Average Day Shift	#	Hourly	4x4	14	0
Average Night Shift	#	Hourly	4x4	11	0
Production					
Total Employed	#	Hourly	4x4	96	51
Average Day Shift	#	Hourly	4x4	24	13
Average Night Shift	#	Hourly	4x4	24	13
Crushing & Hoisting					
Total Employed	#	Hourly	4x4	77	78
Average Day Shift	#	Hourly	4x4	20	20
Average Night Shift	#	Hourly	4x4	19	19
Maintenance					
Total Employed	#	Hourly	4x4	52	38
Average Day Shift	#	Hourly	4x4	14	11
Average Night Shift	#	Hourly	4x4	12	9

Source: JDS (2019)

22.2.2.3 Underground Mining Equipment

Underground mining equipment usage costs are based on the equipment operating hours required to meet the life of mine plan. Equipment usage costs include unit costs ($/hr) for the following elements:

- Maintenance parts;
- Tires;
- Lubricants; and
- Boxes, buckets, and ground engaging tools.

Unit costs for the elements above have been obtained from equipment manufacturer databases and JDS experience. Mobile equipment during the operating period will be owner operated and do not account for any lease, rental, or contractor charges against the equipment.

Equipment replacements and major (mid-life) overhauls are included in the sustaining capital costs.

Table 22-9: Underground Mobile Equipment Cost Summary

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Lateral Development	0.1	1.5	0.05	3%
Production Stoping	3.1	40.2	1.23	70%
Mine General	1.2	15.8	0.48	28%
Total Mining OPEX	**4.4**	**57.5**	**1.76**	**100%**

Source: JDS (2019)

Mobile equipment requirements and operating costs are located in Table 22-10.

Table 22-10: Mobile Equipment Operating Costs (Excluding Fuel)

Equipment	Peak Owner Supply (production period)	Operating Cost ($/hr)
LHD (17t/7.0m^3)	2	134.91
LHD (21t/8m^3)	3	156.08
FEL (15t/5.4m^3)	1	74.77
Surface Truck (60t/35.8m^3)	4	74.26
Jumbo - 2 Boom	1	220.36
Longhole Drill - ITH	5	105.49
Secondary Breakage Drill	2	68.39
Bolter	2	70.04
Cable Bolter	1	98.21
Shotcrete Sprayer	1	12.66
Small Explosives Truck	1	12.53
Large Explosives Truck	2	40.82
Transmixer	1	30.34
Scissor Lift	1	8.26
Fuel/Lube Truck	1	8.56
Mechanics Truck	1	10.74
Electrician Truck	1	10.74
Boom Truck	1	7.77
Grader	1	25.12
Telehandler	1	9.16
Supervisor Truck	6	10.74
Utility Vehicle	6	10.74
Ambulance	1	10.74

Source: JDS (2019)

22.2.2.4 Underground Mining Consumables

Mining consumable usage rates are built up based on the mine plan quantities for development and production activities. Mining consumables include:

- Drill bits and steel;
- Explosives;
- Ground support;
- Piping;
- Electrical cables;
- Ventilation ducting;
- Hoses and fittings;
- Crusher & conveyor parts;
- Hoist and headframe parts; and
- Maintenance tooling.

Consumable unit costs are based on quotations from local suppliers, many of which already provide KDM with open pit consumables. Minor item costs are based on catalog or database values. Ten percent of the base pricing has been added to account for delivery (freight) to site.

Table 22-11: Underground Mining Consumables Summary

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Lateral Development	0.2	3.0	0.09	7%
Production Stoping	1.5	19.4	0.59	48%
Crushing & Hoisting	1.1	14.6	0.45	36%
Mine Maintenance	0.2	3.2	0.10	8%
Mine General	0.0	0.0	0.00	0%
Total Mining OPEX	**3.1**	**40.2**	**1.23**	**100%**

Source: JDS (2019)

22.2.2.5 Underground Fuel Consumption

Underground mining fuel consumption has been built up based on the required equipment operating hours dictated by the mine plan for development or production-based equipment, and annual allowances for support or fixed infrastructure equipment, based on experience at similar operations. Equipment fuel consumption rates have been sourced from local equipment vendors or the list of CANMET-MMSL approved diesel engines for use in underground mines (NRCAN, 2019).

The unit fuel price used in the estimate is US$0.82/litre, inclusive of delivery to site.

Table 22-12: Underground Fuel Cost Summary

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Lateral Development	0.0	0.3	0.01	2%
Production Stoping	1.0	12.4	0.38	61%
Crushing & Hoisting	0.0	0.0	0.00	0%
Mine Maintenance	0.0	0.0	0.00	0%
Mine General	0.6	7.5	0.23	37%
Total Mining OPEX	**1.6**	**20.3**	**0.62**	**100%**

Source: JDS (2019)

Mobile equipment engine and fuel consumption specifications are listed in Table 22-13.

Table 22-13: Mobile Equipment Engine and Fuel Consumption

Equipment Description	Engine Make	Engine Model	CANMET Fuel Consumption
			(l/hr @ 2200RPM)
LHD (17t/7.0m^3)	Volvo	TAD1341VE_369hp	77.7
LHD (21t/8m^3)	Volvo	TAD1344VE_472hp	90.4
FEL (15t/5.4m^3)	CAT	C93	52.6
Surface Truck	CAT	3412e	57.0
Jumbo - 2 Boom	Cummins	QSB4.5_170hp	36.2
Longhole Drill - ITH	Deutz	TCD2013 L04_161hp	33.7
Secondary Breakage Drill	Deutz	BF4M1012C_99.2hp	29.2
Bolter	Detroit Diesel	9043 MU32_148hp	29.2
Cable Bolter	Deutz	TCD2013 L04_161hp	33.7
Shotcrete Sprayer	Detroit Diesel	9043 MU32_173hp	34.3
Small Explosives Truck	Deutz	D914 L06_100hp	21.5
Large Explosives Truck	Deutz	D914 L06_100hp	21.5
Transmixer	Deutz	D914 L06_100hp	21.5
Scissor Lift	Deutz	D914 L06_100hp	21.5
Fuel/Lube Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Mechanics Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Electrician Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Boom Truck	Deutz	D914 L06_100hp	21.5
Grader	Deutz	BF6M1013CP_221hp	43.4

Equipment Description	Engine Make	Engine Model	CANMET Fuel Consumption (l/hr @ 2200RPM)
Mobile Rock Breaker	Detroit Diesel	9043 MU32_148hp	29.2
Telehandler	Perkins	404D-22/C2.2_51hp	12.6
Supervisor Truck	Toyota	1106D-E66TA/C6.6_127hp	34.0
Utility Vehicle	Toyota	1106D-E66TA/C6.6_127hp	34.0
Ambulance	Toyota	1106D-E66TA/C6.6_127hp	34.0

Source: JDS (2019)

22.2.2.6 Underground Power Consumption

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level. Underground mining power includes the power consumption of the underground crushing circuit, headframe, hoists, and surface compressors.

Electricity unit cost is based on a budgetary rate of $0.09/kWh.

Table 22-14: Underground Power Cost Summary

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	M$	M$	$/t	%
Lateral Development	0.0	0.0	0.00	0%
Production Stoping	0.1	1.0	0.03	2%
Crushing & Hoisting	1.7	22.6	0.69	51%
Mine Maintenance	0.0	0.0	0.01	0%
Mine General	1.6	20.7	0.63	47%
Total Mining OPEX	**3.4**	**44.4**	**1.36**	**100%**

Source: JDS (2019)

Power consumption summaries are located in Table 22-15.

Table 22-15: Underground Power Consumption

Power Consumptions	Average Annual	Life of Mine	Unit Cost per tonne Processed	Weighting
	MWh	MWh	kWh/t	%
Mobile Equipment	870	13,920	0.4	2%
Ventilation	6,437	109,436	3.3	16%
Mine Air Cooling	9,491	151,856	4.6	23%
Shaft & Hoisting	19,008	342,145	10.5	51%
Crusher & Conveyor	2,296	32,138	1.0	5%
Dewatering	447	7,596	0.2	1%
Maintenance Facilities	183	2,748	0.1	0%
Miscellaneous Other	394	6,697	0.2	1%
Total Load	**39,126**	**666,536**	**20.4**	**100%**

Source: JDS (2019)

22.2.2.7 Contingency

A 5% contingency has been applied to underground operating costs to account for estimate uncertainties.

22.2.2.8 Mining Cost Metrics

Mine development cost metrics derived from the KDM estimate are summarized below and used to benchmark and validate the mine plan operating costs by third party engineering consultants during the FS preparation. Some metrics apply to capital development activities only and have been summarized here for consolidation purposes.

- Lateral development – US$3,194/m
    - Blended rate of 5.0 x 5.0 m (58%), 5.5 x 5.5 m (26%), 6.0 x 6.0 m (12%), 8.0 x 6.5 m (1%), and 2.0 x 2.0 m (3%).
- Longhole Drilling – US$21.72/m drilled.

22.3 Processing Operating Cost Estimate

The process plant and site infrastructure at Karowe is currently operated by a third-party contractor on a time and materials basis. The processing costs are based on the existing plant yearly operating budget provided by Lucara Botswana and include the following:

- Costs to manage and operate the process plant, audit plant, CRD and FRD facilities and water treatment facilities;
- Site power; and
- Engineering labour for site facilities outside the open pit.

As the underground comes online, the overall site demand for power will increase due to the high load from the shaft. Power costs associated with the underground have been included in the underground OPEX as described in Section 22.2.2.6. The rate that the company pays for power is made up of a combination of fees for actual consumption (total kWhr) and maximum monthly demand (Max kW). As the peak demand of the site increase with the addition of the underground, the fees associated with the maximum monthly demand will also increase leading to an increase in average $/kWhr for the site. Projected power costs included as part of process costs have been increased from the current budget projection of BWP0.91/kWhr to BWP0.95/kWhr for the LOM.

Notwithstanding the changes to the power costs outlined above, there are no material changes anticipated to the process plant as the underground operation comes online, and therefore for the purposes of this study, the existing process OPEX has been extrapolated over the remaining LOM. A summary of costs is provided in Table 22-16.

Table 22-16: Processing OPEX

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed
	M$	M$	$/t
Process OPEX	39.8	833.9	14.88

Source: JDS (2019)

A summary of the Lucara employed processing personnel is provided in Table 22-17. This labour build up does not include any staff employed by the third-party operator.

Table 22-17: Processing Personnel Requirements

Position	Quantity
Process	
Process Manager	1
Technical Superintendent	1
Process Superintendent	1
Process Engineer Technical	1
Process Engineer Production	1
Process Engineer QA/QC	1
Chief Sorter	1
Recovery Supervisor (Recovery Foreman)	4
Plant Metallurgist	1
Laboratory Supervisor	1
Process Clerk of Works	1
Control Room Operator	4
Recovery Operator	10
XRT Operators	15
Operators	4
Senior Diamond Sorter	3
Laboratory Assistant	6
Diamond Sorter	9
Plant Attendants	12
Engineering	
Engineering Manager (Capital Projects Engineer)	1
Asset Integrity Engineer (Mechanical)	1
Maintenance Coordinator - Electrical	1
Maintenance Coordinator - Mechanical	1
Software Technician	1
General Engineering Supervisor	1
Instrument Technician	6
Assistant Software Technician	1
Mechanic	2
Planner	1
Senior Crane Operator	1
Artisan Electrician	6
Artisan Fitter	7
Maintenance Artisan	1
Artisan Boilermaker	4

Position	Quantity
Semi-Skilled Fitter	4
Semi-Skilled Diesel Mechanic	1
Semi-Skilled Boilermaker	2
Crane Operator	1
Semi-Skilled Electrician	2
Documenter	1
Borehole Attendants	5

Source: JDS (2019)

22.4 General and Administrative Operating Cost Estimate

The site General and Administrative (G&A) costs are based on the existing plant yearly operating budget provided by Lucara Botswana and include the costs associated with the following:

- Site finance and administration;
- Human resources;
- Safety, health and environment;
- Mining and mineral resource management; and
- Security.

G&A OPEX in these areas include labour costs, along with all equipment and office supplies, training, fees and permits, and external consultants to support each department as identified by the site.

A summary of costs is provided in Table 22-18.

Table 22-18: G&A OPEX

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed
	M$	M$	$/t
G&A	15.4	323.2	5.77

Source: JDS (2019)

As the operational ramp up of the underground operation coincides with the end of open pit 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 except within security and mineral resource management. Total security personnel and facilities have been increased to provide coverage over the increased area where employees have the potential to interface with diamonds. Mineral resource management personnel have been increased as the operation transitions from third-party mining to owner operated. Table 22-19 outlines the G&A personnel included in the operating cost estimate.

Table 22-19: G&A Personnel Requirement

Position	Quantity
Mine and Mineral Resource Management	
General Manager	1
Technical Manager (Chief Engineer)	1
Administration Assistant*	1
Clerk of Works*	1
Mineral Resources Manager (Chief Geologist)	1
Mine Manager*	1
Geotechnical Geologist	1
Mine Geologist	3
Hydrogeologist	1
Mine technician	1
Surveyor	1
Mining Engineer*	1
Intermediate Mining Engineer*	2
Survey Helper	2
Human Resources	
Human Resources Manager	1
HR Superintendent - IR & Recruitment	2*
HR Superintendent - OD & Housing	1
HR Practitioner - IR & Recruitment	2
HR Practitioner - Organizational Development	1
Systems Administrator (HR Officer)	1
HR Administration Assistant *	1
Finance	
Finance & Administration Manager	1
Mine Accountant	1
Systems Analyst	1
Inventory Coordinator	1
Administration Officer	1
Accounts Supervisor	1
Procurement Officer	1
Administration Assistant - Finance	1
Fuel Administration Assistant	1
Administration Assistant	1
Accounts Assistant	1
Driver	2

Position	Quantity
Senior Office Cleaner	1
Office Cleaner	6
Groundsman	1
Housekeeping & Laundry Attendants*	8
Safety, Health & Environment	
SHE & CR Manager	1
SHE Coordinator	3*
Environmental Officer	2
Community Liaison Officer	1
Safety & Occupational Health Officer	4
Occupational Health Nurse	1
Fire & First Aid Officer	1
Waste Management Operative	1
SHE Administration Assistant*	1
Security	
Security Consultant	1
Security Manager	1
Senior Security Systems Technician	1
Security Superintendent	2
Governance & Intelligence Supt	1
Crime & Intelligence Officer	1
Security Systems Technician L4	2*
Security Systems Technician L3	3
Leaning & Development Officer	1
Team Leader	10*
Governance Officer	1
Administration Officer	2*
Security Officer (Surveillance)	45*
Intelligence Officer	2
Assistant Security Officer (Search & Escort)	71*
Security Technical Operatives	2
Administration Officer*	1

*Indicates new position or increased staffing for existing positions
Source: JDS (2019)

22.5 Cost of Sales and Corporate Operating Cost Estimate

Off-site, in-country corporate costs such as Lucara Botswana management, cost of sales, and direct costs associated with the Clara sales platform have been provided by Lucara. These costs represent costs not directly associated with operating the immediate site, but costs that are still attributable to the Project. The UGP is not anticipated to impact the yearly offsite, in-country costs; as such, the current operational budget provided by Lucara has been extrapolated over the LOM.

A summary of costs is provided in Table 22-20.

Table 22-20: LOM Sales & Corporate Cost

Operating Costs	Average Annual	Life of Mine	Unit Cost per tonne Processed
	M$	M$	$/t
Sales & Corporate Costs	12.2	256.5	4.58

Source: JDS (2019)

23 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in diamond prices and grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

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

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 and Section 22 of this report (presented in 2019 dollars). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Summary of Results

The summary of the mine plan and payable diamonds produced is outlined in Table 23-1. The summaries provided represent the LOM outputs, which include the remaining open pit and current stockpiles, along with the additional value from the development of the underground.

Table 23-1: Life of Mine (LOM) Summary

Parameter	Unit	Value
Ore Processed	Mt	56.0
Mill Average Daily Production	kt/d	7.4
Mill Average Annual Production	Mt	2.7
Average Processing Grade	cpht	13.99
Diamonds Contained	k carats	7,838
Diamonds Recovered	k carats	7,838
Recovery	%	100.0
Initial Capital Cost	US$M	513.7
Sustaining Capital Cost	US$M	208.5
Life of Mine Capital	US$M	722.2

Source: JDS (2019)

Other economic factors include the following:

- Discount rate of 8%;
- Nominal 2019 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- No management fees or financing costs (equity fund-raising was assumed); and
- The model excludes all pre-development and sunk costs up to the start of detailed engineering for the underground development (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.) and any costs incurred to the end of 2019 for the open pit operations.

23.2 Assumptions

Table 23-2 and Table 23-3 outline the diamond prices and exchange rate assumptions used in the economic analysis. The diamond prices have been provided by Lucara and are based on historical information, market assessments and statistical analysis of the anticipated size distribution supported by data sets derived from the existing operations (Section 19).

Table 23-2: Economic Assumptions

Item	Unit	Value
NPV Discount Rate	%	8
Annual Escalation	%	0
BWP:US$ FX	BWP:US$	10.6
ZAR:US$ FX	ZAR:US$	14

Source: JDS (2019)

Table 23-3: Baseline Diamond Prices

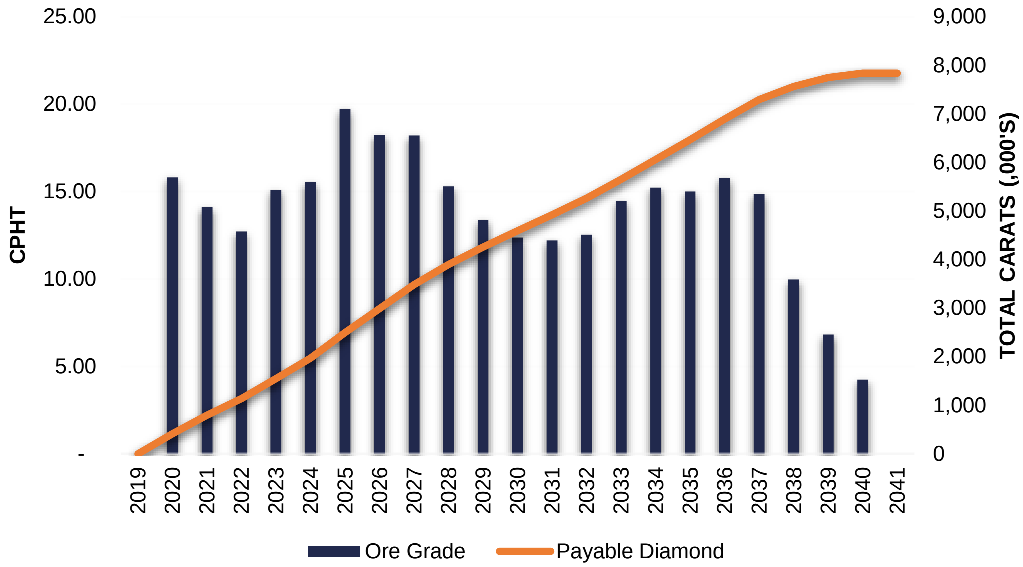
Unit	2020	2021	2022	FS
North	222	222	222	222
Centre	323	329	349	349
EM/PK(S)	618	705	741	777
M/PK(S)	513	578	604	631

Source: JDS (2019)

Efforts have been made to provide realistic estimates for diamond prices and exchange rates based historical performance, current sales information and potential future markets. It should be noted that diamond prices and exchange rates are based on many complex factors and there are no reliable long-term predictive tools.

Figure 23-1 shows the grade and total carats recovered during the LOM.

Figure 23-1: Grade and Carat recovery by Year



Source: JDS (2019)

23.3 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate value of the potential project economics. The methodology for the tax calculation was provided by Lucara Botswana for incorporation into the model, the completed tax model was then reviewed by Lucara Botswana. The tax model contains the following assumptions:

- Income Tax: Annual tax rate = 70 – 1500/x:
  - Where x is the profitability ratio, given by taxable income as a percentage of gross income;
  - Where the calculated rate shall not be less than the company rate of 22%; and
  - Net Losses, incurred in years of high CAPEX expenditures, can be deferred to future years to offset tax liabilities.
- VAT modeled with a net zero impact due to expected VAT credits and status as exporter; and
- Withholding taxes on foreign consulting services included as a capital cost within the owner's CAPEX

Total taxes for the Project are estimated at the amount of $936 M.

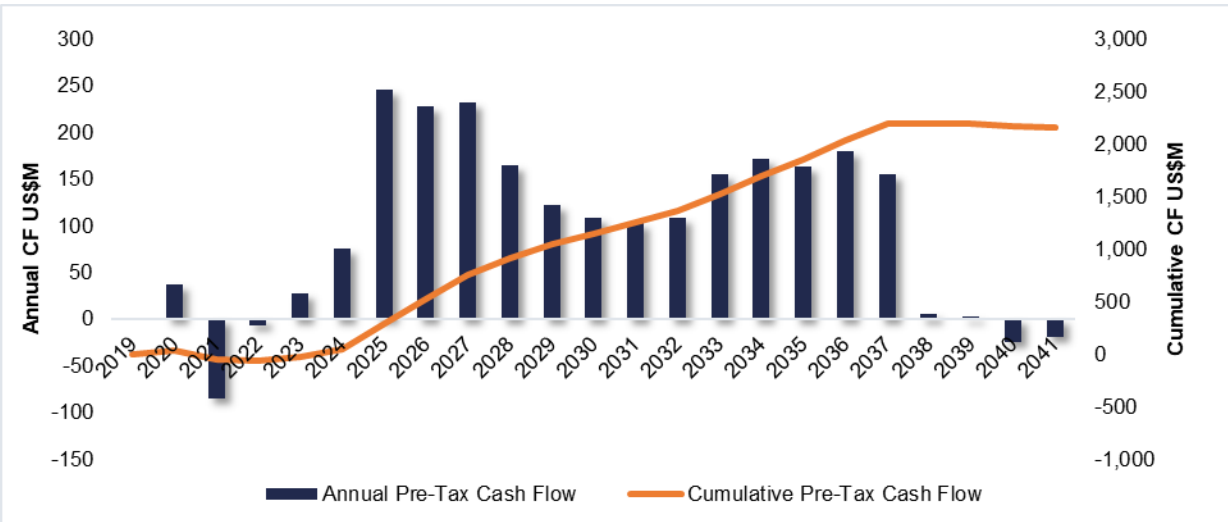
23.4 Royalties

The KDM is subject to a royalty payable to the Botswana Government of 10% of all sales. Estimated royalty payments amount to $525 M over the remaining LOM.

23.5 Results

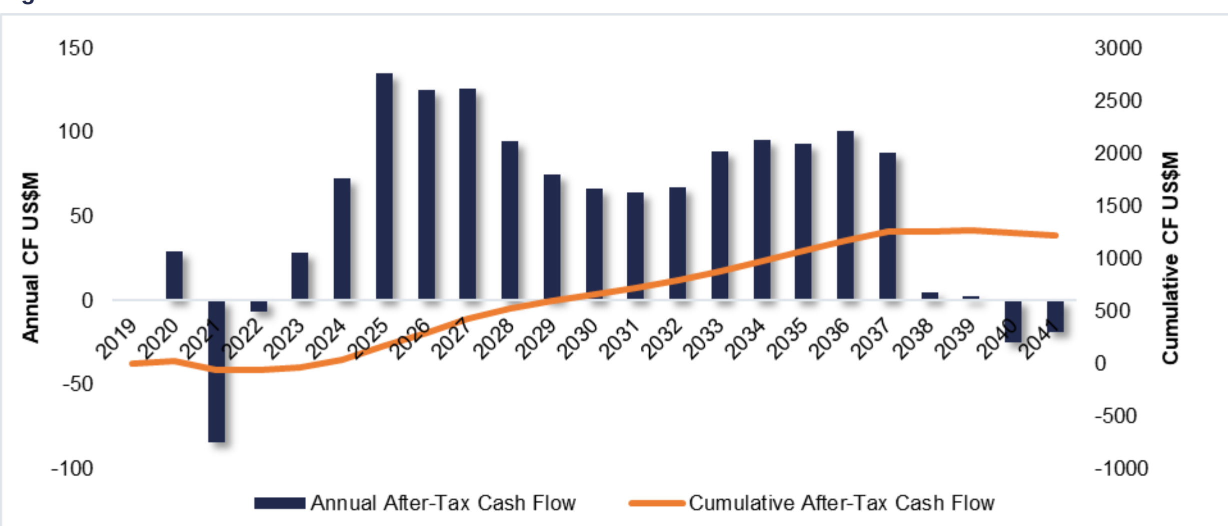
The Karowe LOM, including the development of the UGP, is economically viable with an after-tax net present value using an 8% discount rate ($NPV_{8\%}$) of $532 M using the diamond prices described in Section 23.2. Figure 23-2 and Figure 23-3 shows the projected cash flows, and Table 23-4 summarizes the economic results of the Karowe Underground Project.

Figure 23-2: Pre-Tax Cash Flows



Source: JDS (2019)

Figure 23-3: After-Tax Cash Flows



Source: JDS (2019)

The after-tax break-even average diamond price is approximately US$414/carat, based on the LOM plan presented herein. This is the diamond price at which the Project $NPV_{8\%}$ discount rate is zero.

The life of mine all-in sustaining cost (AISC) is US$397/ct. The straight AISC cost is calculated by adding the sales & corporate, royalty, operating, and capital and closure costs together and dividing by the total payable carats.

The LOM economic model does not calculate a meaningful Internal Rate of Return (IRR), as capital costs are partially offset by operating revenue during the years they are incurred. An underground specific economic model was developed to evaluate the incremental value provided by the development of the project. In the UG only evaluation, the Project showed pre- and after-tax IRR's of 20.8% and 16.0% respectively.

Table 23-4: Economic Results - LOM Model

Parameter	Unit	Pre-tax Results	After-tax Results
$NPV_{0\%}$	US$M	2,156.7	1,220.4
$NPV_{8\%}$	US$M	945.3	535.4
IRR	%	N/A	N/A
Payback period	Production years	2.8	2.8

Source: JDS (2019)

23.6 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -20% to +20%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the diamond prices were evaluated at a +/- 20% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as diamond prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

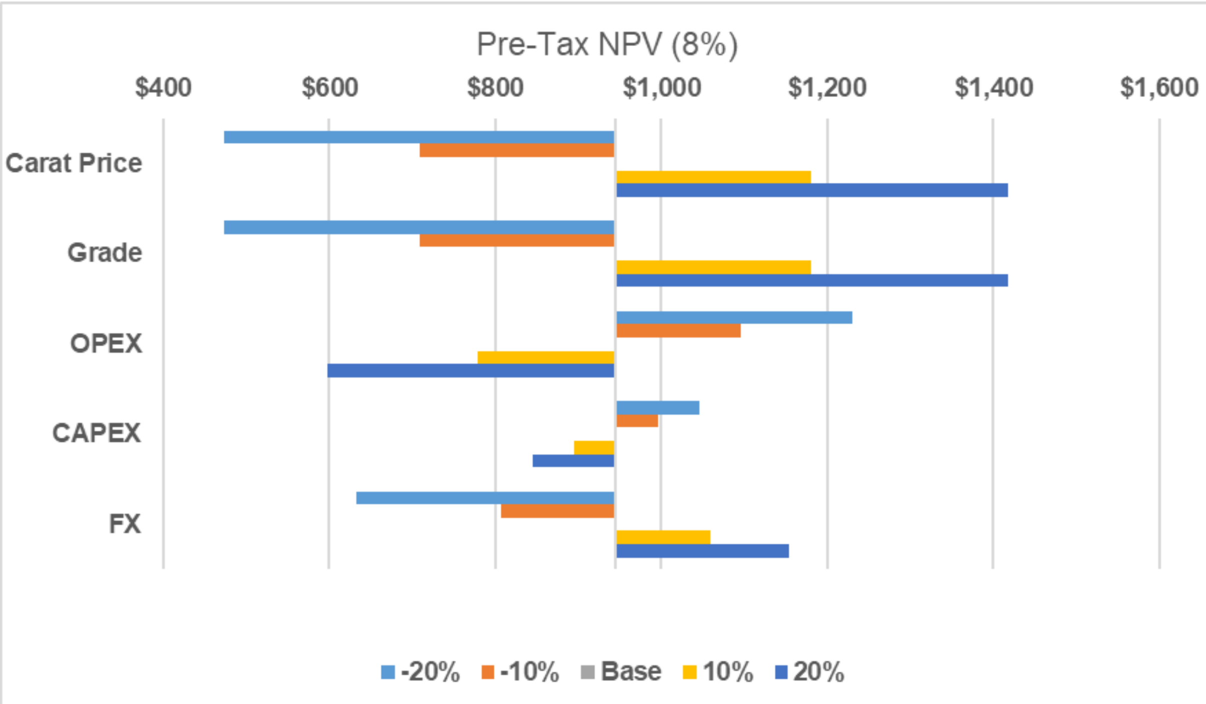
Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the Project is most sensitive to diamond prices grade. The Project showed the least sensitivity to capital costs. Table 23-5 and Figure 23-4 show the results of the sensitivity tests, while Table 23-6 shows the NPV at various discount rates.

Table 23-5: Sensitivity Results (NPV @ 8%, IRR)

Variable	Pre-tax $NPV_{8\%}$ (M$)		
	-20% Variance	0% Variance	20% Variance
CAPEX	1,046	945	845
OPEX	1,230	945	598
Diamond Price or Grade	474	945	1,417

Source: JDS (2019)

Figure 23-4: Sensitivity Results - Tornado Plot



Source: JDS (2019)

Table 23-6: Pre-Tax NPV Discount Rate Sensitivity

Discount Rate	0%	5%	8%	10%	15%
NPV (M $)	2,157	1,266	945	786	512

Source: JDS (2019)

The cash flow for the Project is shown in Figure 23-5.

Figure 23-5: LOM Cash Flow

			2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
PRICE			0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Carat Price	US$/Carat	**670**	0	520	553	611	630	666	772	765	758	725	697	674	678	686	709	716	721	729	680	379	457	609	0
Escalation	Annual %	**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%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
MINE PRODUCTION																									
Waste Mined	**k tonnes**	**13,432**	**-**	**3,990**	**2,536**	**2,607**	**2,483**	**1,589**	**228**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**			
Open Pit Waste	k tonnes	12,682	-	3,985	2,413	2,380	2,087	1,589	228	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UG Waste	k tonnes	751	-	4	123	227	396	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore Mined	**k tonnes**	**49,969**	**-**	**3,774**	**3,207**	**2,521**	**2,895**	**3,539**	**3,788**	**2,597**	**2,596**	**2,592**	**2,589**	**2,587**	**2,587**	**2,588**	**2,590**	**2,591**	**2,592**	**2,593**	**1,741**	**-**	**-**	**-**	**-**
OP Ore Mined	**k tonnes**	**16,507**	**-**	**3,774**	**3,207**	**2,521**	**2,628**	**3,077**	**1,299**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
UG Ore Mined	**k tonnes**	**33,462**	**-**	**-**	**-**	**-**	**267**	**462**	**2,489**	**2,597**	**2,596**	**2,592**	**2,589**	**2,587**	**2,587**	**2,588**	**2,590**	**2,591**	**2,592**	**2,593**	**1,741**	**-**	**-**	**-**	**-**
Ore Grade	**cpht**	**14.83**	**-**	**0.14**	**0.13**	**0.13**	**0.15**	**0.15**	**0.18**	**0.19**	**0.18**	**0.15**	**0.13**	**0.12**	**0.12**	**0.12**	**0.14**	**0.15**	**0.15**	**0.16**	**0.16**	**-**	**-**	**-**	**-**
Contained Carats	**k carats**	**7,409**	**-**	**532**	**424**	**332**	**443**	**516**	**689**	**481**	**475**	**393**	**339**	**313**	**310**	**318**	**370**	**391**	**386**	**410**	**287**	**-**	**-**	**-**	**-**
MILL FEED																									
Ore Throughput	**k tpd**	**7.4**	**-**	**7.3**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**7.5**	**5.8**	**-**
Ore Processed	**k tonnes**	**56,029**	**-**	**2,632**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,097**	**-**
Ore Grade	**cpht**	**13.99**	**-**	**15.81**	**14.11**	**12.73**	**15.10**	**15.54**	**19.72**	**18.25**	**18.22**	**15.31**	**13.38**	**12.37**	**12.22**	**12.54**	**14.47**	**15.22**	**15.00**	**15.77**	**14.87**	**9.97**	**6.82**	**4.25**	**-**
Contained Carats	**k carats**	**7,838**	**-**	**416**	**381**	**344**	**408**	**420**	**532**	**493**	**492**	**413**	**361**	**334**	**330**	**339**	**391**	**411**	**405**	**426**	**401**	**269**	**184**	**89**	**-**
Stockpile	k carats	185	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	96	89	-
Recovery	**%**	**100%**	**0%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**100%**	**0%**
Recovered Carats	**k carats**	**7,838**	**-**	**416**	**381**	**344**	**408**	**420**	**532**	**493**	**492**	**413**	**361**	**334**	**330**	**339**	**391**	**411**	**405**	**426**	**401**	**269**	**184**	**89**	**-**
REVENUE																									
Payable	%	**100%**	0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
	k carats	**7,838**	-	416	381	344	408	420	532	493	492	413	361	334	330	339	391	411	405	426	401	269	184	89	-
Revenue	**US$M**	**5,253**	**-**	**216**	**211**	**210**	**257**	**279**	**411**	**377**	**373**	**300**	**252**	**225**	**224**	**232**	**277**	**294**	**292**	**310**	**273**	**102**	**84**	**54**	**-**
Royalties	% of Value	**10%**	0%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	0%
	US$M	**525**	-	22	21	21	26	28	41	38	37	30	25	23	22	23	28	29	29	31	27	10	8	5	-
Corporate Costs - Botswana	US$M	**196**	-	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	-
Cost of Sales	US$M	**60**	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-
Net Revenue	**US$M**	**4,471**	**-**	**183**	**177**	**177**	**219**	**239**	**358**	**327**	**323**	**258**	**214**	**190**	**189**	**197**	**237**	**253**	**250**	**267**	**234**	**80**	**64**	**37**	**-**
OPERATING COSTS																									
Mining - OP	US$/tonne mined	**4.71**	-	4.12	4.67	4.88	4.88	4.82	5.57	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M	**136**	-	32.0	26.2	23.9	23.0	22.5	8.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining - Rehandle	US$/tonne rehandled	**1.16**																							
	US$M	**14**	-	0.2	0.4	0.7	0.3	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.2	3.3	3.3	2.6	-
Mining - UG	US$/tonne mined	**8.72**	0	-	-	-	-	-	14.13	11.69	11.16	9.66	9.00	7.78	6.91	6.92	6.90	6.90	6.90	6.91	8.78	-	-	-	-
	US$M	**286**	-	-	-	-	-	-	35.2	30.3	29.0	25.0	23.3	20.1	17.9	17.9	17.9	17.9	17.9	17.9	15.3	-	-	-	-
Processing	US$/tonne processed	**14.88**	-	14.34	14.71	14.72	14.79	14.85	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.97	14.55	-
	US$M	**834**	-	37.7	39.7	39.7	39.9	40.1	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	40.4	30.5	-
G&A	US$/tonne processed	**5.77**	-	5.08	5.12	5.13	5.19	5.20	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	6.04	4.63	-
	US$M	**323**	-	13.4	13.8	13.9	14.0	14.0	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	9.7	-

			2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Subtotal - OPEX	**US$/tonne processed**	**28.42**	**-**	**385.53**	**356.44**	**347.84**	**343.24**	**341.16**	**446.54**	**387.48**	**31.78**	**30.32**	**29.69**	**28.51**	**27.67**	**27.69**	**27.67**	**27.67**	**27.67**	**27.68**	**27.10**	**22.22**	**22.22**	**20.40**	**-**
	US$M	**1,593**	**-**	**83**	**80**	**78**	**77**	**77**	**100**	**87**	**86**	**82**	**80**	**77**	**75**	**75**	**75**	**75**	**75**	**75**	**73**	**60**	**60**	**43**	**-**
NET OPERATING INCOME	**US$M**	**2,879**	**0**	**99**	**97**	**98**	**142**	**162**	**257**	**240**	**238**	**176**	**134**	**113**	**114**	**122**	**162**	**178**	**176**	**192**	**160**	**20**	**4**	**-6**	**0**
CAPEX																									
Initial			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1100 - Mining Surface Infrastructure	US$M	**2.1**	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1200 - Mining Underground Equipment	US$M	**27.2**	-	-	-	-	-	27	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1300 - Mining Underground Infrastructure	US$M	**10.9**	-	-	-	1	10	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1400 - Mining Underground Development	US$M	**70.7**	-	-	-	11	37	23	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1500 - Mining Underground Systems	US$M	**35.3**	-	0	6	12	16	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1600 - Capitalized Underground Production	US$M	**15.4**	-	0	0	1	4	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1700- Shaft Sinking & Infrastructure	US$M	**160.1**	-	12	104	34	8	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2000 - Bulk Earthworks	US$M	**18.8**	-	5	2	4	7	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3000 - Process Plant	US$M	**0.1**	-	-	-	-	-	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4000 - Tailings & Mine Waste Management	US$M	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5000 - Onsite Infrastructure	US$M	**5.9**	-	1	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
6000 - Buildings & Facilities	US$M	**1.6**	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7000 - Off-Site Infrastructure	US$M	**19.6**	-	4	13	3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
8000 - Project Indirects	US$M	**47.7**	-	7	17	13	6	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9000 - Owners Costs	US$M	**46.9**	-	15	8	8	8	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining																									
Sustaining	US$M	**208.5**	-	9	8	8	7	5	11	12	5	11	11	5	12	14	6	5	12	13	5	14	0	19	19
Subtotal - CAPEX	**US$M**	**670.8**	**-**	**57**	**162**	**95**	**102**	**82**	**11**	**12**	**5**	**11**	**11**	**5**	**12**	**14**	**6**	**5**	**12**	**13**	**5**	**14**	**0**	**19**	**19**
Contingency	US$M	**51.4**	-	*5*	*20*	*10*	*11*	*6*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL CAPEX	**US$M**	**722.2**	**-**	**62**	**182**	**105**	**113**	**87**	**11**	**12**	**5**	**11**	**11**	**5**	**12**	**14**	**6**	**5**	**12**	**13**	**5**	**14**	**0**	**19**	**19**
NET PRE-TAX CASH FLOW	**US$M**	**2,157**	**0**	**37**	**-85**	**-7**	**28**	**75**	**246**	**228**	**233**	**165**	**123**	**108**	**103**	**108**	**156**	**172**	**164**	**180**	**155**	**6**	**3**	**-25**	**-19**
Cumulative Cash Flow	**US$M**		**0**	**37**	**-48**	**-54**	**-26**	**49**	**295**	**523**	**756**	**921**	**1,044**	**1,152**	**1,255**	**1,363**	**1,519**	**1,692**	**1,856**	**2,036**	**2,191**	**2,197**	**2,200**	**2,175**	**2,157**
TAXES	%	43%	0%	22%	0%	0%	0%	22%	45%	45%	46%	43%	39%	39%	37%	38%	43%	44%	43%	44%	44%	22%	22%	0%	0%
TAXES	**US$M**	**936**	**0**	**8**	**0**	**0**	**0**	**3**	**111**	**103**	**107**	**70**	**48**	**42**	**38**	**41**	**68**	**77**	**71**	**79**	**68**	**1**	**1**	**0**	**0**
NET AFTER-TAX CASH FLOW	**US$M**	**1,220**	**0**	**29**	**-85**	**-7**	**28**	**73**	**135**	**125**	**126**	**94**	**75**	**66**	**64**	**67**	**88**	**96**	**93**	**101**	**88**	**5**	**3**	**-25**	**-19**
Cumulative Cash Flow	**US$M**		**0**	**29**	**-56**	**-63**	**-35**	**38**	**174**	**299**	**424**	**519**	**593**	**660**	**724**	**791**	**880**	**976**	**1,069**	**1,169**	**1,257**	**1,261**	**1,264**	**1,239**	**1,220**

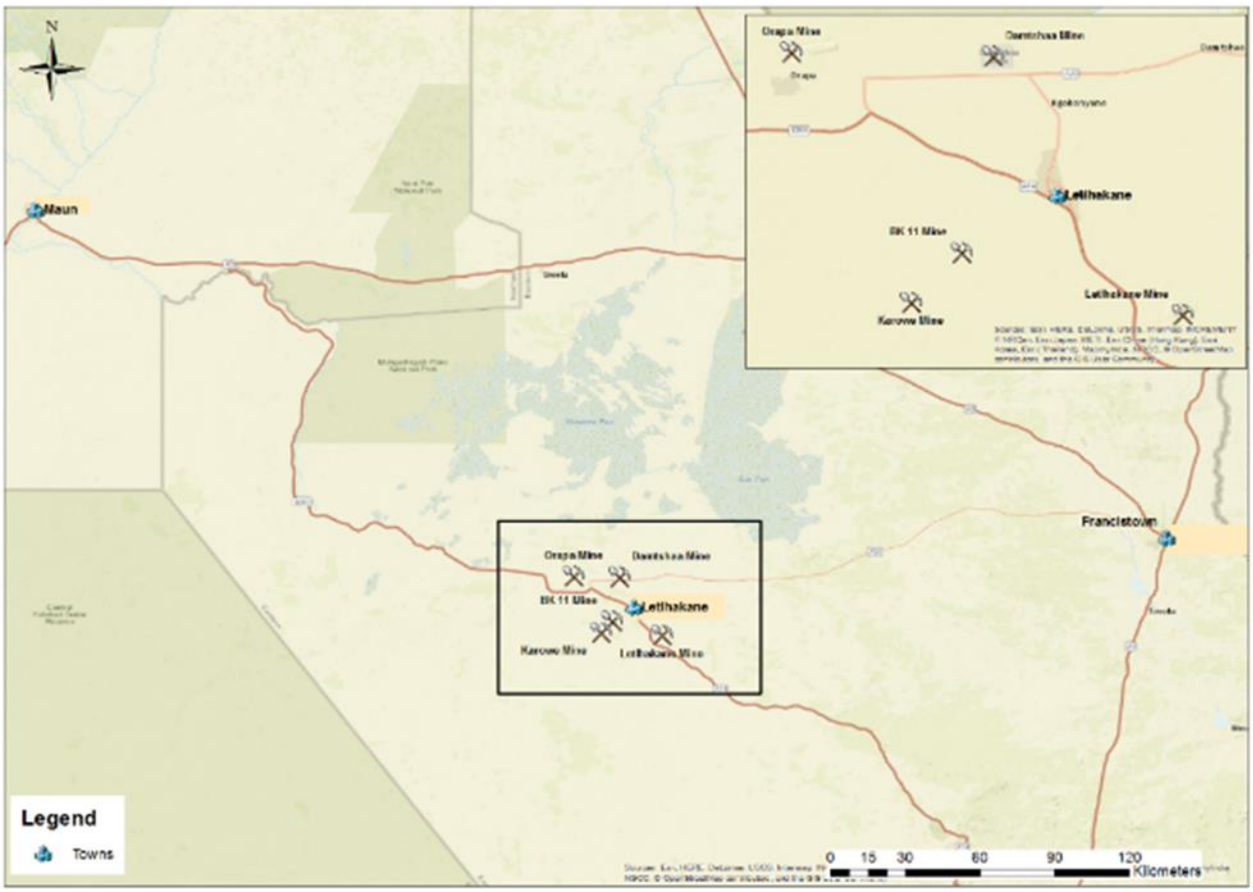
Source: JDS (2019)

24 Adjacent Properties

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

The Karowe Mine is based on the AK6 kimberlite pipe, which is part of the Orapa kimberlite field. Nine kimberlite pipes in this field are either operating mines or have been mined in the past. Current major adjacent diamond mines are shown in Figure 24-1 and summary details are provided in Table 24-1. Orapa is the second largest commercially exploited kimberlite in the world. The Letlhakane Mine produces diamonds of very high quality. The Damtshaa Mine is based on four relatively low-grade kimberlites.

Figure 24-1: Locations of Major Diamond Mines Proximal to the Karowe Mine



Source: Oberholzer et al. (2017)

Table 24-1: Summary Information for the nearby Orapa, Letlhakane and Damtshaa Mines

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

Source: Anglo American Ore Reserves and Mineral Resources Report (2016)

25 Other Relevant Data and Information

25.1 Project Execution Plan

25.1.1 Introduction

The Karowe Project Execution Plan (PEP) describes the project development strategies that were considered for the FS capital cost estimate and project schedule. The PEP is meant to provide the future framework for organizing the engineering, procurement, and construction. The Execution Plan will also serve as a guide in:

- Promoting safety in design, construction, and operations in order to succeed;
- Negotiating contracts with suppliers, contractors, and engineers with proven track records in Botswana; and
- Planning the project execution in a way that allows the project to leverage the existing site workforce and maximizes local labour as much as possible when external contractors are required.

Although the Execution Plan provides guidance for executing the Project, the planning stage will evaluate alternate execution strategies and other opportunities that add value overall. This may include items such as variations to portions of the execution strategy (i.e. Engineering, Procurement and Construction Management (EPCM), Engineering, Procurement and Construction (EPC), Engineering, Procure and Supply (EPS), etc.) or, inclusion of owner resources for smaller scopes of work.

25.1.2 Project Development Schedule

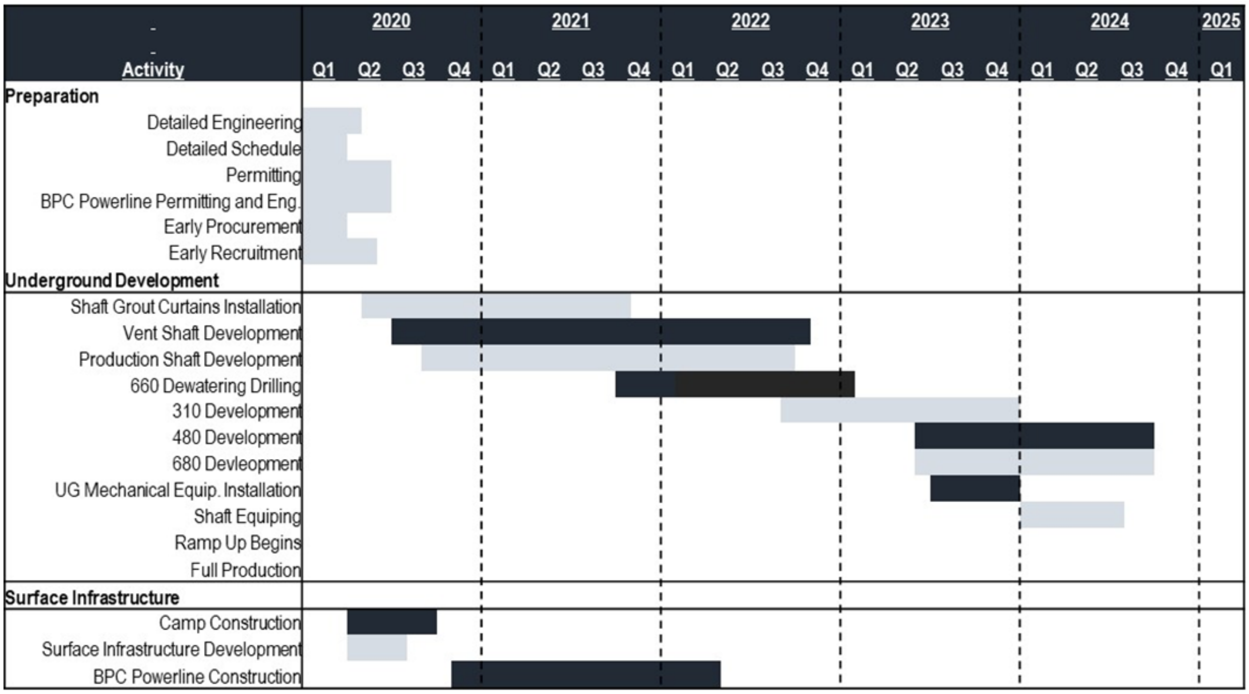
The overall development period for the Project is estimated to be approximately five years, from the start of detailed engineering to the underground reaching over 60% production capacity.

The critical path of the schedule runs through the following activities:

- EPCM contract formation;
- Shaft engineering and procurement;
- Shaft grout curtain;
- Shaft sinking;
- Main dewatering program;
- Lateral development of the 310 Level; and
- Drawbell development.

Activities completed in 2020 will include detailed engineering and permitting, site preparation, camp development and surface infrastructure construction, implementation of the grout curtain and the completion of the pre-sink for the both shafts. Work will continue to ramp up in 2021 as the sinking of the shaft progresses, dewatering activities progress and the BPC powerline is constructed. The shaft sinking will reach the extraction level at the end of 2022, when lateral development will begin. Level development will be complete mid-2024, and production will start to ramp up in Q4 2024, with the underground reaching full production in Q1 2025. Additional details are provided in Figure 25-1.

Figure 25-1: Karowe UGP Execution Schedule

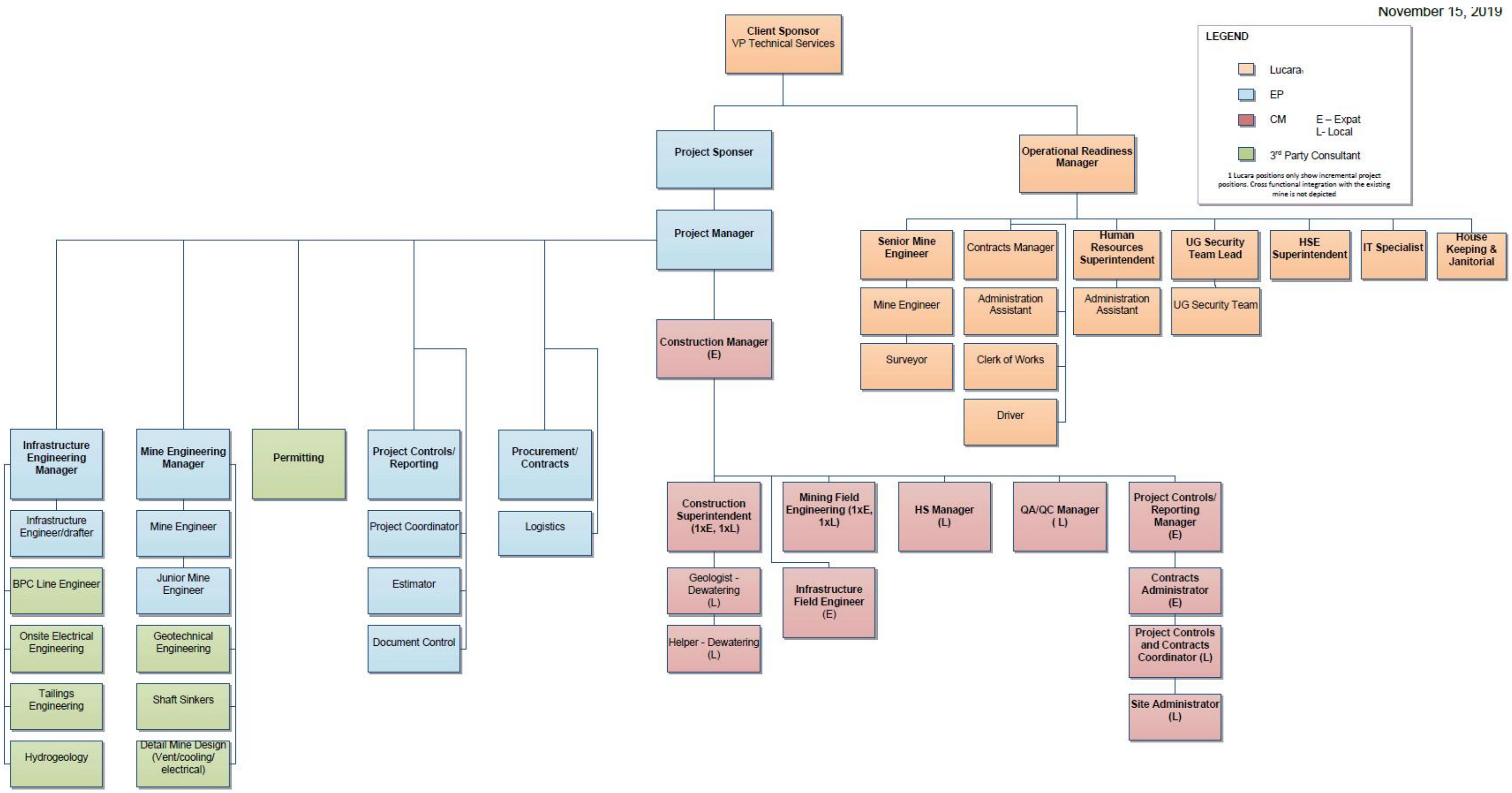


Source: JDS (2019)

25.1.3 Project Management

The Project Management Team ("PM Team") will be an integrated team including the owner's personnel, the EPCM contractor, and various engineering contractors. The PM Team will oversee and direct all engineering, procurement, and construction activities for the Project. Figure 25-2 presents the preliminary project organization chart for both the engineering and construction phases of the Project.

Figure 25-2: Organizational Structure



Source: JDS (2019)

25.1.4 Engineering

The general engineering execution strategy for the Project will be to utilize multiple engineering firms with specialized knowledge of their assigned scope. Coordination of engineering interfaces and overall management of engineering schedule and deliverables will be the responsibility of the EPCM project manager or infrastructure and mining leads. The following major engineering contract packages have been identified for the Project:

- Detailed engineering and procurement for the shafts;
- Geotechnical characterization;
- Detailed engineering of CRD and FRD facilities;
- Detailed design of the underground infrastructure and utilities (electrical distribution, ventilation and cooling, crushing and conveying); and
- Hydrological characterization, water balance, and water management systems including dewatering wells.

25.1.5 Construction

During the construction Phase, the Project Manager (or their designate) will be responsible for the development and construction areas. The designated EPCM Construction Manager and Lucara Operational Readiness Manager will closely coordinate site activities, to maintain project efficiency and minimize the impacts to the current operation. The main objectives of the construction execution strategy will include:

- Execute all activities with a goal of zero harm to people, assets, the environment, or reputation;
- Strive to eliminate process, operational and maintenance safety hazards;
- Meet or exceed environmental regulatory and permit requirements to minimize impact;
- Cultivate an atmosphere of positive social impact in the surrounding communities;
- Maximize the involvement of the existing site workforce;
- Utilize local labour as much as possible where external contractors are required;
- Identify and remove barriers that affect project progress; and
- Recognize, identify and communicate outstanding achievements during construction and commissioning of the Project.

26 Interpretations and Conclusions

26.1 Risks

The Project Risk Register was prepared at the FS-level based on direct interviews and inputs from the disciplines leads: geotechnical, hydrogeology, mining, shafts sinking and CRD/FRD management. The Risk Register also took into account a re-assessment of the risks identified at the previous PEA stage. The FS Risk Register is presented in Table 26-1.

Table 26-1: FS Risks Register - Main Project Risks

No	Risk Statement	Risk Category	Cause/Consequence	Mitigation	Risk Status under Mitigation
1	Work Permits and Certification of foreign workers and technical staff. ▪ The risk is related to potential delays, especially in the early stages of the Project, associated with the approval by Botswana Government of the work permits, licensing and certification for foreigners.	Schedule Risk	▪ Government insistence on hiring local labour, and therefore not granting permission for external skills in favour of training local skills. ▪ Bureaucracy in the processing of work permits applications. ▪ Delays in the delivery of work permits and certifications will put the shaft development schedule at risk, which has further consequences for the whole execution schedule of the UG mine development.	High level engagement with Botswana Government.	Medium
2	Delay in the procurement of hoist and shaft infrastructure.	Schedule Risk	▪ Refurbishment of the hoist that will be reused is required in advance of the development of the shafts. ▪ Failure to commit to early procurement for the refurbishment of the hoist will compromise the development schedule of the shafts.	Commitment to early procurement.	Medium
3	Capacity and availability of local contractors and suppliers to provide construction support services and equipment.	Schedule Risk	▪ Competitive market is expected locally by other mining projects in the vicinity of Karowe; expected high local demand for various construction support services (transport, fuel supply, customs services, aggregates, food supply, etc.) and construction equipment. ▪ Delays in the development of the shafts due to lack of local resources. ▪ It may be necessary to bring skills and resources from surrounding countries due to the issues relating to importing materials and/ or work permits etc.	Commitment to early Logistics Plan and Procurement.	Medium
4	Delay in the open pit dewatering program.	Schedule Risk	▪ As of September 2019, the Immediate Dewatering Acceleration Program (IDAP) was behind schedule due to a combination of factors related to procurement, delivery and staffing. ▪ Further delay in the progress of the open pit dewatering program will impact the development of the shafts as well as the overall dewatering plan for the period 2020-2032.	Fast track the open pit dewatering work. New pumps were installed, and dewatering efficiency has since improved. Real-time dewatering management software in place for close monitoring of dewatering targets vs actual.	Medium
5	Shaft sinking through weak / wet sandstone aquifer zones (Ntane and Mosolotsane formations).	Technical Risk – Construction.	▪ Sinking of the shafts will intersect weak, low competency carbonaceous formations and permeable zones with high pressure flow velocities. ▪ Construction challenges; slowdown in sinking rates meter/day; impact on shafts development schedule.	Design includes pre-grouting to seal exposure of the shafts to high pressure groundwater inflows during construction. Early commitment to pre-drilling and mobilization of contractor for grouting;	Medium
6	Failure (during stoping and drawdown) of the weak host rock formation (Tlapana) that surrounds the kimberlite pipe.	Technical Risk - Geotechnical	▪ Weakness of layers in the host rock (Tlapana Formation). ▪ Sudden failure could cause major inflow of host rock into the excavation followed by air blast through tunnels and shafts.	Design is based on leaving in place a 15 m high-strength kimberlite ring (barrier) against the weak host rock. The protective kimberlite ring is to be recovered at the end of LOM. Operational mitigation: to maintain the muck pile against the walls of the kimberlite pipe.	Low
7	Failure to preserve the 15 m kimberlite ring during drilling of blasting holes.	Technical Risk - Mining	▪ Long drill holes, lack of drill hole alignment accuracy, deviation near the walls of the kimberlite ring.	Drill & Blast design. Monitoring.	Low

No	Risk Statement	Risk Category	Cause/Consequence	Mitigation	Risk Status under Mitigation
			▪ Drill holes may accidentally hit the walls of the kimberlite ring, thus weakening its integrity with consequential risk of partial collapse of the kimberlite ring.		
8	Brow sloughing and large fragmentation / oversize ore material.	Technical Risk - Mining	▪ Long drill holes over widely spaced drilling horizon. ▪ Oversize material will affect draw control and block draw points.	Design flexibility allows reduction of length of drill holes and the addition of drilling sublevels if needed.	Low
9	Presence of methane and other gases in the underground mine. ▪ Some incidents of methane gas emissions were reported during drilling in the open pit and gas bubbling in sumps. These might be indicators of potential presence of gas during underground mining.	Technical Risk - Mining	▪ Shale can promote methane gas production. ▪ Levels of methane gas emissions can trigger threshold for mine classification as gaseous mine under applicable regulations, with consequences for equipment specification. ▪ Mine equipment has not been specified as flameproof, nor is flameproof equipment available in the sizes selected for the mine plan.	Further data acquisition and investigation of gas emissions. Since mining is to take place inside the 15 m thick kimberlite ring, and because the draw is located in the granite formation, exposure of mining operations to gas from the shale formations has a low likelihood.	Medium
10	Large areas of unsupported and hanging kimberlite mass rock as blasting retreats vertically.	Technical Risk - Mining	▪ Blasting sequence. ▪ Large blocks could be liberated from the unsupported kimberlite mass and could create draw control issues and blockages. ▪ Sudden failure of the unsupported kimberlite could create air blast.	Likelihood low due to the high-strength and high-density of the kimberlite. Mining to proceed while minimizing gap by management of muck pile. Monitoring extensometers.	Low
11	Excessive salinity of deep water pumped from the granite formation between 2032-2045.	Technical Risk – Hydrogeology	▪ Expected TDS concentrations in the deep water to be pumped from the granite aquifer are 25,000 mg/L. ▪ Mixing of this water at the process plant raw water tank with other sources of water from dewatering could result in exceeding the limit of 4,000 mg/L TDS for acceptance of delivering Karowe excess water to the local water consumer, with the consequence of no other possibility to dispose of water above 4,000 mg/L TDS. ▪ Re-use of high salinity water will impact the process plant water circuit.	Design includes grouting of the granites as far as practically feasible to reduce ingress of saline waters Maximum abstraction rate of deep saline water has been established (30 to 40 m^3/hr) so that mixing with other sources can comply with the limit of 4,000 mg/L TDS for acceptance by the local water consumer of Karowe excess water. A better understanding of the granite formation should be acquired in the next step in particular with the grouting of the fractured granite.	Medium
12	Overflow in the underground tunnel below 310 L of excess water resulting from the 1 in 100-year storm event between 2026-2040. FS management of excess water for the 1:100-year condition is based on: 1) Use of full capacity of the pipeline to the local water consumer; 2) Capacity of on-ramps paddocks to retain 40,000 m^3; 3) New surface settling pond 40,000 m^3; 4) Storage capacity in tunnels 35,000 m^3; and 5) Shut down of return water from TSF to process plant for eight days.	Operational Risk – Water Management	▪ Failure of on-ramp paddocks to retain up to 40,000 m^3 of storm water for the 1 in 100-year event. ▪ Flooding of tunnels below 310 L and equipment with consequential disruption of mining operations.	Maintain contingency to collect and to pump storm water from the open pit. Further validation of the on-ramp paddocks system. Operational water management plan and procedures to be developed in a next step including contingency measures.	Medium
13	Insufficient temporary water storage capacity available at the new slimes storage facility to allow for shutting down the return of water to the process plant during the 1 in 100-year storm event.	Technical Risk – Water Management.	▪ FS management of storm water in the 1 in 100-year condition requires shut down of water return from the slimes storage facility to the process plant for up to eight consecutive days. ▪ Failure to shut down return of water to the process plant due to lack of available storage capacity will create local overflow at the slimes storage	Commitment to developing a site wide integrated operational water management plan and procedures including contingency measures.	Medium

No	Risk Statement	Risk Category	Cause/Consequence	Mitigation	Risk Status under Mitigation
			facility with consequential risk to the integrity of the walls of the slimes storage facility.		
14	Deficit of water supply to the process plant.	Operational Risk – Water Management	▪ During 2032-2040, dewatering volumes are scheduled to decrease below process plant water demand. ▪ Shortage of water supply to the process plant is not an option.	New water supply wellfield to be developed and to be permitted in the south Karowe area.	Low
15	Decision by local water consumer to no longer accept Karowe excess water for reasons other than the limit of 4,000 mg/L TDS.	Operational Risk	▪ Although a local water off-take Agreement is in place between local water consumer and Karowe for evacuating Karowe excess water through the pipeline to local consumer, many possible reasons could take place in the future for the local water consumer to stop acceptance of this water. ▪ Under cancellation by the local water consumer of the agreement, Karowe site water balance would then become unmanageable.	Alternatives to sending Karowe excess water to local water consumer are available for evaluation in a next step as contingency measures. Among the options at this point: water supply to other neighbouring water consumer mine; artificial groundwater recharge far away from Karowe UG mine; offsite evaporation.	Medium
16	Neighbour farmers to face higher pumping costs due to the regional lowering of the water table as a result of the Karowe open pit dewatering program.	Technical & Community Risk	▪ Extended influence on the regional groundwater of the Karowe open pit dewatering over a 20 years period ▪ Consequence of having to pay compensations to neighbour farmers who need irrigation water.	A regional groundwater flow and water supply model to be developed in a next step and integrated with the local Karowe mine dewatering model in order to provide information about the radius of the dewatering influence and the cumulative impacts on groundwater uses by farmers and by other mining operations in the area. Sustained community engagement.	Medium
17	Local pollution of groundwater.	Technical Risk – Water Management	▪ Arsenic is currently detected in the monitoring wells of the existing TSF at concentrations slightly exceeding WHO standard for drinking water. ▪ Further seepage from the new slimes storage facility would increase the contaminated plume which could result in Public Health issues related to potential uses of groundwater outside of Karowe property.	FS showed very slow travelling rate of the arsenic plume in the order of 150 m over 100 years. A transition layer of sand has been included in the wall design to prevent piping (i.e. open paths for leakage of slimes water). Operational procedures are such that water in the slimes storage facility will be pumped off to the process plant so that to minimize contact of water with the porous outer walls. Consolidation of the very fine material at the bottom of the new slimes storage facility to create an impervious barrier.	Low
18	Failure to raise the walls of the new slimes storage facility at the time intervals specified by the design.	Operational Risk – Infrastructure	▪ The design of the new slimes storage facility is based on successive raises of the walls with lifts of 5 m for each raise. ▪ Failure to achieve timely construction of the successive raises will create insufficient storage capacity to receive slimes during operations; over-topping, and internal and upstream failure of walls.	Rigorous monitoring of the elevations reached by the deposited slimes during operation. Enforcement of the wall raising schedule specified by the design.	Low
19	Failure to re-evaluate the draw plan during mining operations.	Operational risk - Mining	▪ Lack of follow-up by the operational mining team. ▪ The day-to-day draw plan is an important factor for the performance of the recovery of ore and control of dilution.	Operations procedures and incentive policy.	Low
20	Build-up of water at the top muck pile during mechanical failure / downtime of material handling equipment and risk of flooding in the extraction area following re-start of extraction.	Technical risk - Mining	▪ During downtime of material handling equipment, the muck pile must be kept moving to maintain mixing of dry / wetter materials and prevent potential accumulation of water. ▪ Minimum draw shall continue even if no material handling is taking place.	Design includes availability of temporary storage to achieve a minimum draw of six buckets per day, for four days, thus allowing for maintaining movement and mixing of the muck pile.	Low

No	Risk Statement	Risk Category	Cause/Consequence	Mitigation	Risk Status under Mitigation
			▪ Accumulated water would flow down to the extraction area when extraction to re-start and shake the muck pile above with consequential possibility of flooding and further disruption of mining operations.		
21	Damages to large size diamonds during blasting.	Technical risk - Mining	▪ Blasting plan, holes diameter and powder factor. ▪ Substantial loss of value resulting from damages to large size diamonds.	Design based on similar powder factor as in current open pit operations and smaller holes diameter; diamond damages expected to be consistent with open pit operations. Secondary fragmentation of the muck pile may allow for the powder factor to be reduced further with potential reduction of damages.	Medium
22	Confidence in the mining method – "bottom-up" Long Hole Shrinkage (LHS.)	Technical risk - Mining / Geotechnical	▪ The "bottom-up" LHS mining method is unprecedented in diamond mining to the scale being considered for the present project. ▪ Absence of other similar applications at the scale of the present project creates technical uncertainties.	The proposed mining method takes advantage of and benefits from the unique high density and high strength of the Karowe kimberlite. The high-density, high strength of the Karowe kimberlite allows for mining inside a protective 15 m kimberlite ring that resolves stability issues with the weak host rock formations around the pipe. The mining method is supported by strong back-up of data from extensive drilling and geotechnical modelling. Third Party review of the bottom up LHS method was conducted as part of the FS, showing that more conventional SLC mining method would share similar technical uncertainties due to the specifics of Karowe geological formations and hydrogeological conditions.	Medium
23	Subject to a written exemption that can be obtained under the Botswana mining regulation, the installation of the main ventilation fans should be at the surface. Since the FS design is based on installing the main fans in the underground mine (as opposed to installing at the surface), the risk is related to not obtaining the necessary exemption.	Regulatory Risk	▪ Article 548 of Botswana Mining Regulation. ▪ Design change if exemption not obtained.	Early engagement by Lucara with the mining regulator for applying for the exemption.	Medium

Source: JDS (2019)

26.2 Opportunities

Several opportunities have been identified during the FS that could improve project economics, reduce risk or improve execution. Table 26-2 highlights some of the more significant opportunities that will be explored during value and detailed engineering stages planned for late 2019, early 2020.

Table 26-2: Identified Project Opportunities

Opportunity	Explanation
Re-design of the OP with new block model	The open pit mine plan has not yet been adjusted for the new 2019 mineral resource estimate block model. Based on JDS's review, additional open pit carats and/or reduced waste and higher value ore brought forward are all expected outcomes of the re-optimization and design of the open pit. The re-design of the open pit is expected to be completed in Q1 2020.
Reduced shaft cost and duration	Several cost saving initiatives are currently underway to decrease the contraction duration of the shaft, save material costs, defer non-critical capital expenses and lower the overall cost of the shafts.
Kimberlite skin optimization	The buffer zone planned to be temporarily left behind to hold back carbonaceous shale dilution needs to be optimized. Currently the skin extends from the granite up through the top of the UG stope. This is not likely necessary, and the skin may be abide to be stopped at the mudstone/carbonaceous shale contact therefore freeing up more tonnes of high value EMPKS earlier without significant dilution risk.
Electrification of UG equipment	The UG LHD fleet could be run as tethered electric units to reduce ventilation costs and potentially lead to automation.
Stockpile optimization	As the open pit mine plan is updated the surface stockpile schedule will be revised, potentially adding higher value mill feed material sooner.
Some upper development CAPEX could be delayed and put into sustaining CAPEX	A full detailed CAPEX review will be conducted in early 2020 and will consider ways of deferring or reducing CAPEX. An example is the build-up of the full construction team currently is planned to start in 2020 while many of the positions will not be needed until later in the year.
Mining below 310 L down to 250 L, INF to 60 masl and open	Approximately 1.8 Mt of ore, mainly high-value EMPKS is below the currently planned mine between 250 masl and 310 masl. This indicated resource has not been included in the UG FS but would add high-value material early if the shafts are deepened or additional material at the end of the mine life. There are over 300,000 carats in this zone.
UG mining of North and or Central lobes	Potential incremental value may be obtained in the UG mine by extracting the north and central zones below the open pit. This opportunity will be pursued later in the mine life as the lower value of the North and Central lobes will not help the project economics if they are mined early.
Increased production rate after 2029	Once drilling and blasting is complete, production from UG can be increased to >3.1 Mt/a

Opportunity	Explanation
Recovery of exceptional diamonds	If exceptional diamonds continue to be recovered at the historical rate ($250 M in value projrct to date), economics improve significantly. The recovery of exceptional diamonds was not included in the FS economic analysis.

Source: JDS (2019)

27 Recommendations

The Karowe UG Project is economically viable and detailed engineering and financing should both be pursued.

Early works not identified in the FS capital cost estimate should be conducted as a priority including:

- Advance risk mitigation exercises (Dec 2019) ($50k):
  - Work permits, concrete and local contractor supply investigation;
- Optimize open pit, design and schedule ($70k);
- Start value engineering review and optimization of the UG and OP mine plans ($120k);
- Start detailed shaft and mine engineering ($350k);
- Start detailed cost estimation and scheduling with a shaft sinking contractor ($60k); and
- Start procurement on critical path items definition and sourcing ($50K).

28 References

Armstrong, R. and Venter, J., 2007. Feasibility Mining Geotechnical Evaluation of the AK6 Kimberlite Pipe. Unpublished consulting report prepared by SRK Consulting, June 2007.

Angrove, B., 2019. “Karowe Comminution Tests – BL0449”, unpublished letter by Base Metallurgical Laboratories Ltd., September 2019.

Barnett, W.P., 2007. AK6 Country Rock Model. Unpublished internal memo, DeBeers Consolidated Mines, March 2007.

Bosma, P., 2008. AK6 Mineral Resource Classification review. Unpublished internal Memo, De Beers Group Services (Pty) Ltd. - MRM

Boteti Exploration (Pty) Ltd, 2007. Environmental Impact Assessment for the Proposed AK 06 Diamond Mine: EIA Report Geoflux; Gaberone; September 2007 pp.81-86.

Bush, D.E., 2006. AK6 Phase 1 – Mineral Resource Estimate. Unpublished internal report, De Beers Mineral Resource Management (MRM).

Bush, D.E., 2008a. Review of the AK6 Mineral Resource Estimate. Unpublished report prepared by Z-Star for De Beers Group Services (Pty) Ltd.

Bush, D.E., 2008b. Technical Note: Mineral resource statement for AK6; from surface to 750m depth. Unpublished report prepared by Z-Star for De Beers Group Services (Pty) Ltd.

Bush, R., Mossop, D. and Terbrugge, P., J., 2017. Geotechnical and Hydrogeological Review of the Karowe Open Pit. Unpublished consulting report by SRK Consulting, January 2017.

Coates, J.N.M, Davies, J., Gould, D., Hutchins, D.G., Jones, C.R., Key, R.M., Massey, N.W.D., Reeves, C.V., Stansfield, G. and Walker, I.R., 1979. The Kalatraverse One Report. Geological Survey Department, Botswana, Bulletin 21.

Department of Water Affairs and Forestry, 1999. National Water Act, 1998 (ACT NO. 36 OF 1998), Regulations on use of water for mining and related activities aimed at the protection of water resources. *Goverment Notice No. 704*. South Africa: Goverment Gazette (Vol. 408, No. 20119).

Digby Wells Associates, 2018. Environmental Management Plan and Mine Closure Plan for Karowe Diamond Mine: Rehabilitation and Closure Plan; Report # BTM5403; December 2018.

Digby Wells Environmental, 2019. Summary Social Impact Assessment: Karowe Diamond Mine. BTM5399 (August 2019).

Doll, A., 2019. “Technical Memorandum: Lucara Karowe AG Mill Evaluation” unpublished Technical Memo by Alex G Doll Consulting Ltd., October 2019.

Doll, A., 2019. “AGD_LucaraMemo_201910” unpublished test results by Alex G Doll Consulting Ltd., October 2019.

EBS, 2012. Boteti mine: Environmental & social due diligence. Unpublished consulting report prepared by Environmental Business Strategies (Pty) Ltd., October 2012.

EBS, 2017. Boteti mine: Environmental & social due diligence. Unpublished consulting report prepared by Environmental Business Strategies (Pty) Ltd.

EBS, 2017. Environmental, Social & Governance Compliance Assessment 2016 Karowe Mine, Botswana EBS; Johannesburg; February 2017

Ekkerd, J. and Ruest, M., 2008. Geotechnical Slope Stability Study. DeBeers Consolidated Mines. April 2008.

Geoflux, 2007. Environmental Impact Assessment of the proposed AK6. Unpublished consulting report prepared by Geoflux (Pty) Ltd.

Geoflux, 2013. Closure and Rehabilitation Plan Report. Unpublished consulting report prepared by Geoflux (Pty) Ltd., August 2013.

Geoflux, 2014. EMP for the update of Karowe Diamond Mine. Unpublished consulting report prepared by Geoflux (Pty) Ltd.

Geoflux, 2018. Karowe Diamond Mine, Closure and Rehabilitation Plan, 2017 Liability Update. Unpublished consulting report prepared by Geoflux (Pty) Ltd.

Geoflux, 2016. Environmental Management Plan Update (EMP) for Karowe Diamond Mine Geflux; Gaberone; 2016.

Geoflux, 2008. EIA for the proposed AK06 mine on behalf of De Beers Prospecting Botswana Geoflux; Gaberone; 2008.

GTD Consulting, 2018. Valuation of BK72 EMPKS Rough Diamond Sample. Unpublished consulting report prepared by GTD Consulting Limited for Lucara Diamond Corp., March 2018.

Hanekom, A., Stiefenhofer, J., Robey, J.v.A., 2006. Geology of the AK6 kimberlite – current knowledge and progress update. Unpublished internal report, De Beers Geological Survey (DGBS), Mineral Resource Management (MRM).

Itasca, 2015. Updated Numerical flow model for Karowe Mine. Unpublished consulting report prepared by the Itasca Consulting Group.

Lipton, I. T., 2001. Measurement of bulk density for resource estimation in mineral resources and ore reserve estimation. In: The AusIMM Guide to Good Practice. Ed. AC Howards, p. 57 – 66.

Lowther, R.J., Olivier, L., Lett, J.L., Brunton, I., 2016. Implementation of a Surface-based Hydraulic Fracturing Program to Sucessfully Propagate a Large Cave through Hard, Competent, Near-Surface Rock Masses to Achieve Breakthrough. Sydney, Australia, May 9-11, 2016.

Lynn, M., Nowicki, T., Valenta, M., Robinson, B., Gallagher, M., Robin, B. and Sexton, J., 2014. Karowe Diamond Mine, Botswana, NI 43-101 Independent Technical Report, Effective Date 31 December 2013, posted to www.sedar.com.

McGeorge, I., Lynn, M., Ferreira, J., Croll, R., Blair, D., Morton, K., (MSA), 2010. NI 43-101 Technical Report on the Feasibility Study for the AK6 Kimberlite Project, Botswana, Effective Date 12August 2010, unpublished technical report prepared by MSA Geoservices (Pty) Ltd for Lucara Diamond Corp., posted to www.sedar.com.

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

MSC17/006R, 2017. AK6 South Lobe pipe shell update. Unpublished consulting memo prepared by Mineral Services Canada Inc. for Lucara Diamond Corp.

MSC18/005R, 2018. Update of the AK6 South Lobe geological model, Karowe Diamond Mine, Botswana. Unpublished consulting report prepared by Mineral Services Canada Inc. for Lucara Diamond Corp.

MSC17/038R, 2017. Core photo review and revisions to the EM/PK(S) boundary: preliminary results. Unpublished consulting memo prepared by Mineral Services Canada Inc. for Lucara Diamond Corp.

MSC18/004R, 2018. AK6 geological model update. Unpublished consulting memo prepared by Mineral Services Canada Inc. for Lucara Diamond Corp.

MSC18/005R, 2018. Update of the AK6 South Lobe geological model, Karowe Diamond Mine, Botswana. Unpublished consulting report prepared by Mineral Services Canada Inc. for Lucara Diamond Corp.

Natural Resources Canada (NRCAN), 2019. The Canadian Standards Association (CSA) CANMET-MMSL approved diesel engines for use in underground mines and confined locations such as tunnels. Viewed Aug 20 2019, <https://www.nrcan.gc.ca/mining-materials/resources/approved-diesel-engines/8180>, last modified 2019-05-24Tait, M.A., Maccelari, M.G., 2008: Update to the three-dimensional geological model of the AK6 kimberlite, Botswana: modifications following the 2008 front end engineering design (FEED) review by ATD. Unpublished internal report, De Beers Group Services (Pty.) Ltd.

Nowicki, T.E., Armstrong, J., Fourie, L.J.H., 2018. 2018 Technical Report for the Karowe Mine: Updated Mineral Resource Estimate, Effective Date 7 August 2018, unpublished technical report prepared by Mineral Services Canada Inc. for Lucara Diamond Corp., posted to www.sedar.com.

Oberholzer, G.J., Blackham, N.G.C, Cox, J.A., Thompson, J.J., Morton, K.L, Nowicki, T., Armstrong, J., 2017. NI 43-101 Technical Report on the Preliminary Economic Assessment of the Karowe Diamond Mine Underground Project, Effective Date 31 October 2017, unpublished technical report prepared by Royal HaskoningDHV for Lucara Diamond Corp., posted to www.sedar.com.

Opperman, A, and van der Schyff, W., 2007. Phase 2 Three-Dimensional Geological Modelling of the AK6 kimberlite, Botswana. Unpublished consulting report prepared by Golder Associates Africa (Pty) Ltd for De Beers Group Services (Pty).

Schulze, D.J., 1987. Megacrysts from alkalic volcanic rocks. In: Nixon, P.H. (ed) Mantle xenoliths. John Wiley & Sons, Chichester, United Kingdom, pp 433-451

Scott Smith, B.H., Nowicki, T.E., Russell J.K., Webb K.J., Mitchell, R., Hetman, C.M., Harder, M., Skinner E.M.W. and Robey, JvA., 2013. Kimberlite terminology and classification. D.G. Pearson et al. (eds), Proceedings of the 10th International Kimberlite Conference, v2, Special Issue of the Journal of the Geological Society of India, p1-17.

Scott Smith, B.H., Nowicki, T.E., Russell J.K., Webb K.J., Mitchell, R., Hetman, C.M., Robey, JvA., 2018. A Glossary of Kimberlite and Related Terms. Published by Scott-Smith Petrology Inc., North Vancouver, BC, Canada. Part 1 – 144 pp; Part 2 – 59 pp; Part 3 – 56 pp.

Shibistova, A., Simons, P., 2019. "First Inspection Report" for Karowe UG Kimberlite and Host Rock. Unpublished internal report by TOMRA Sorting GmbH.

SiVest Investment Pty Ltd., 2010. Environmental Management Plan for the Karowe Diamond Mine Rev. SiVest; Gaberone.

SRK, 2019. Update of the AK6 geological model in support of a Mineral Resource update for the 2019 Karowe Mine Underground Feasibility Study; unpublished report prepared by SRK Consulting (Canada) Inc. for Lucara Botswana (Pty) Ltd.

Stiefenhofer, J., 2007. Geology of the AK6 kimberlite July 2007. Unpublished internal report, De Beers Mineral Resource Management (MRM).

Stiefenhofer, J, Hanekom, A., 2005. Geology of the A/K6 kimberlite – current knowledge and progress update. Unpublished internal report, De Beers Mineral Resource Management (MRM).

Tait, M.A. and Maccelari, M.G., 2008. Update to the three dimensional geological model of the AK6 kimberlite, Botswana: modifications following the 2008 front end engineering design (FEED) review by ATD. Unpublished internal report, De Beers Group Services (Pty) Ltd.

Terbrugge, P.J. and Mossop, D.H., 2017. Cut 2 Optimisation at Karowe. Unpublished consulting memo prepared by SRK Consulting for Boteti Mining.

The South African Bureau of Standards (SABS), October 27, 1998. South African National Standard (SANS) 10286 (ICS 73.020). SABS 0286:1998: Code of Practice - Mine Residue. Pretoria, South Africa: The South African Bureau of Standards

29 Units of Measure, Abbreviations and Acronyms

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
Acfm	actual cubic feet per minute
ACK	apparent coherent kimberlite
ALT	active layer thickness
ALT	active layer thickness
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BC	Block cave
BD	bulk density
BPC	Botswana Power Corporation
Bt	billion tonnes
BTU	British thermal unit
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
cfm	cubic feet per minute
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

Symbol / Abbreviation	Description
c/s	carats per stone
c/t	carat per tonne
cpht	carats per hundred tonnes
Cr	chromium
ct	carat
cts	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
DMS	dense media separation
dmt	dry metric ton
DTC	diamond trading company
DWT	dead weight tonnes
EA	environmental assessment
EIA	environmental impact assessment
EIS	environmental impact statement
ELC	ecological land classification
EMP	environmental management plan
EM/PK(S)	Eastern magmatic/pyroclastic kimberlite
ERD	explosives regulatory division
EWR	enhanced winter road
FAR	Fresh air raise
FEL	front-end loader
FRD	Fine residue deposits
ft	foot
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

Symbol / Abbreviation	Description
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GPa	gigapascal
gpm	gallons per minute (us)
GTZ	glacial terrain zone
GW	gigawatt
h	hour
ha	hectare
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m2)
ha	hectare
HG	high grade
HK	hypabyssal kimberlite
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
IRR	internal rate of return
ITH	In the hole hammer
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
kg	kilogram
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

Symbol / Abbreviation	Description
km/h	kilometres per hour
km^2	square kilometre
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
LG	low grade
LHS	Long hole shrinkage
LHOS	Long hole open stoping
LOM	life of mine
m	metre
M	million
m/min	metres per minute
m/s	metres per second
m^2	square metre
m^3	cubic metre
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	metres above sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum

Symbol / Abbreviation	Description
mbs	metres below surface
Mct	million carats
MCRP	Mine Closure and Rehabilitation Plan
mg	milligram
mg/L	milligrams per litre
MIDA	microdiamond
min	minute (time)
ML	Mining license
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMSIM	metamorphosed massive sulphide indicator minerals
mo	month
MPa	megapascal
M/PK(S)	magmatic/pyroclastic kimberlite
MWR	Mega watts of refridgeration
MSC	Mineral Services Canada Inc.
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
Nm^3/h	normal cubic metres per hour
NMD	
NQ	drill core diameter of 47.6 mm
NRC	natural resources Canada
OP	open pit
OPEX	Operating expenses
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PEA	preliminary economic assessment
PFK	processed fine kimberlite

Symbol / Abbreviation	Description
PFS	preliminary feasibility study
PGE	platinum group elements
PK	pyroclastic kimberlite
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RC	reverse circulation
RMR	rock mass rating
R/O	Reverse osmosis
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
RVK	resedimented volcaniclastic kimberlite
s	second (time)
SEP	Stakeholder engagement plan
SLC	Sub level cave
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SG	specific gravity
SRC	Saskatchewan Research Council
SRK	SRK Consulting Inc.
stns	stones
st/kg	stones per kilogram
st/t	stones per metric tonne
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
TCR	total core recovery
TFFE	target for further exploration

Symbol / Abbreviation	Description
TSF	tailings storage facility
t/h	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
UCS	unconfined compressive strength
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VEC	valued ecosystem components
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
WBT	Wet bulb temperature
w/w	weight/weight
wk	week
wmt	wet metric ton
WRSF	waste rock storage facility
XRT	X-Ray transmission
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