TECHNICAL REPORT ON THE KEMESS PROJECT, NORTH-CENTRAL BRITISH COLUMBIA

CENTERRA GOLD INC.

Technical Report pursuant to NI 43-101

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Cautionary Note Regarding Forward-Looking Information

All statements, other than statements of historical fact contained or incorporated by reference in this document, which address events, results, outcomes or developments that Centerra expects to occur are, or may be deemed to be, forward looking information or forward-looking statements within the meaning of certain securities laws, including the provisions of the Securities Act (Ontario) and the provisions for "safe harbor" under the United States Private Securities Litigation Reform Act of 1995 and are based on expectations, estimates and projections as of the date of this document. Such forward-looking information involves risks, uncertainties and other factors that could cause actual results, performance, prospects and opportunities to differ materially from those expressed or implied by such forward-looking information. Forward-looking statements are generally, but not always, identified by the use of forward-looking terminology such as "assume", "believes", "continue", "encouraged", "estimate", "expect", "future", "ongoing", "plan", "potential", "project", "target", "update" or "upside", or variations of such words and phrases and similar expressions or statements that certain actions, events or results "may", "could", "would" or "will" be taken, occur or be achieved or the negative connotation of such terms. Such statements include but may not be limited to: the estimation of mineral resources, including inferred mineral resources, at Kemess and the potential of eventual economic extraction of minerals from the project; the identification of future mineral resources at the project; Centerra's ability to convert existing mineral resources into categories of mineral resources or mineral reserves of increased geological confidence; life of mine estimates; future exploration potential; timing and scope of future exploration (brownfields or greenfields); the project design, including the location of infrastructure and the proposed open pit and underground mine plans; the project development timeline to production including future phases of the project and development and construction of and production at the project, including the possibility of constructing either or both of an open pit and underground mines; the timing of and future prospects for exploration and any expansion of the project, including upside associated with the project's land package; the future success of Kemess including the mining methods and the possibility of constructing either or both of an open pit and underground mines; the potential for expanding the mineral resources; the potential for identifying additional mineralization in areas of intercepts and conceptual areas for extension and expansion; any potential synergies between the Kemess project and satellite deposits, if any; the ability of the existing infrastructure at Kemess to lower execution risk for the project and the possibility that any additional infrastructure will complement it; and the expectation that a leach plant will increase gold recovery.

Centerra cautions that forward-looking statements are necessarily based upon a number of factors and assumptions that, while considered reasonable by the Company at the time of making such statements, are inherently subject to significant business, economic, technical, legal, political and competitive uncertainties and contingencies, which may prove to be incorrect, include but are not limited to: there being no significant disruptions affecting the activities of the Company whether due to extreme weather events and other or related natural disasters, labour disruptions, supply disruptions, power disruptions, damage to equipment or otherwise; permitting and development of the project being consistent with the Company's expectations; political and legal developments in British Columbia and Canada being consistent with its current expectations; continued compliance with financial covenants in Centerra's credit agreement that is secured by certain assets used at the Kemess Project, Centerra's access to cash flow from its subsidiaries; and changes to taxation laws; risks arising from the streaming agreement with Triple Flag Mining Finance Bermuda Ltd.; the accuracy of the current mineral resource estimates of the Company; certain price assumptions for gold and copper and foreign exchange rates; the Company's future relationship with Indigenous groups being consistent with the Company's expectations; and inflation and prices for diesel, natural gas, fuel oil, electricity and other key supplies being approximately consistent with anticipated levels. Known and unknown factors could cause actual results to differ materially from those projected in the forward-looking statements and undue reliance should not be placed on such statements and information.

Market price fluctuations in gold, copper, and other metals, as well as increased capital or production costs or reduced recovery rates may render ore reserves containing lower grades of mineralization uneconomic and may ultimately result in a restatement of mineral reserves. The extent to which mineral resources may ultimately be reclassified as proven or probable mineral reserves is dependent upon the demonstration of their profitable recovery. Economic and technological factors, which may change over time, always influence the evaluation of mineral reserves or mineral resources. Centerra has not adjusted mineral resource figures in consideration of these risks and, therefore, Centerra can give no assurances that any mineral resource estimate will ultimately be reclassified as proven and probable mineral reserves.

Mineral resources are not mineral reserves, and do not have demonstrated economic viability, but do have reasonable prospects for economic extraction. Indicated mineral resources are sufficiently well defined to allow

geological and grade continuity to be reasonably assumed and permit the application of technical and economic parameters in assessing the economic viability of the resource. Inferred mineral resources are estimated on limited information not sufficient to verify geological and grade continuity or to allow technical and economic parameters to be applied. Inferred mineral resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as mineral reserves. There is no certainty that mineral resources of any category can be upgraded to mineral reserves through continued exploration.

Centerra's mineral reserve and mineral resource figures are estimates, and Centerra can provide no assurances that the indicated levels of gold or copper will be produced, or that Centerra will receive the metal prices assumed in determining its mineral reserves. Such estimates are expressions of judgment based on knowledge, mining experience, analysis of drilling results, and industry practices. Valid estimates made at a given time may significantly change when new information becomes available. While Centerra believes that these mineral reserve and mineral resource estimates are well established, and the best estimates of Centerra's management, by their nature mineral reserve and mineral resource estimates are imprecise and depend, to a certain extent, upon analysis of drilling results and statistical inferences, which may ultimately prove unreliable. If Centerra's mineral reserve or mineral reserve estimates for its properties are inaccurate or are reduced in the future, this could have an adverse impact on Centerra's future cash flows, earnings, results, or operations and financial condition.

There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Forward-looking statements are provided for the purpose of providing information about management's expectations and plans relating to the future. All of the forward-looking statements made in this document are qualified by these cautionary statements and those made in our other filings with the securities regulators of Canada and the United States including, but not limited to, those set out in the Company's latest 40-F/Annual Information Form and Management's Discussion and Analysis, each under the heading "Risk Factors", which are available on SEDAR+ (www.sedarplus.ca) or on EDGAR (www.sec.gov/edgar). The foregoing should be reviewed in conjunction with the information, risk factors and assumptions found in this document.

Centerra disclaims any intention or obligation to update or revise any forward-looking statements, whether written or oral, or whether as a result of new information, future events or otherwise, except as required by applicable law.

Cautionary Note to U.S. Investors

Centerra prepares its disclosure in accordance with the requirements of securities laws in effect in Canada. Unless otherwise indicated, all Mineral Resource and Mineral Reserve estimates included in this document have been prepared in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101") and the Canadian Institute of Mining, Metallurgy and Petroleum (the "CIM") - CIM Definition Standards on Mineral Resources and Mineral Reserves, adopted by the CIM Council, as amended (the "CIM Standards"). NI 43-101 is a rule developed by the Canadian Securities Administrators, which established standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects. Mining disclosure in the United States was previously required to comply with SEC Industry Guide 7 ("SEC Industry Guide 7") under the United States Securities Exchange Act of 1934, as amended. The U.S. Securities and Exchange Commission (the "SEC") has adopted final rules, to replace SEC Industry Guide 7 with new mining disclosure rules under sub-part 1300 of Regulation S-K of the U.S. Securities Act ("Regulation S-K 1300") which became mandatory for U.S. reporting companies beginning with the first fiscal year commencing on or after January 1, 2021. Under Regulation S-K 1300, the SEC now recognizes estimates of "Measured Mineral Resources", "Indicated Mineral Resources" and "Inferred Mineral Resources". In addition, the SEC has amended its definitions of "Proven Mineral Reserves" and "Probable Mineral Reserves" to be substantially similar to international standards.

Investors are cautioned that while the above terms are "substantially similar" to CIM Definitions, there are differences in the definitions under Regulation S-K 1300 and the CIM Standards. Accordingly, there is no assurance any mineral reserves or mineral resources that Centerra may report as "proven mineral reserves", "probable mineral reserves", "measured mineral resources", "indicated mineral resources" and "inferred mineral resources" under NI 43-101 would be the same had Centerra prepared the mineral reserve or mineral resource estimates under the standards adopted under Regulation S-K 1300. U.S. investors are also cautioned that while the SEC recognizes "measured mineral resources", "indicated mineral resources" and "inferred mineral resources" under Regulation S-K 1300, investors should not assume that any part or all of the mineralization in these categories will ever be converted into a higher category of mineral resources or into mineral reserves. Mineralization described using these terms has a greater degree of uncertainty as to its existence and feasibility

than mineralization that has been characterized as reserves. Accordingly, investors are cautioned not to assume that any measured mineral resources, indicated mineral resources, or inferred mineral resources that Centerra reports are or will be economically or legally mineable.

Non-GAAP Measures

This document contains the following non-GAAP financial measures: all-in sustaining costs per ounce sold on a by-product basis, sustaining capital expenditures, and non-sustaining capital expenditures. These financial measures do not have any standardized meaning prescribed by GAAP and are therefore unlikely to be comparable to similar measures presented by other issuers, even as compared to other issuers who may be applying the World Gold Council ("WGC") guidelines, which can be found at http://www.gold.org.

All-in sustaining costs on a by-product basis per ounce sold include adjusted operating costs, the cash component of capitalized stripping costs, corporate general and administrative expenses, accretion expenses, and sustaining capital, net of copper and silver credits. The measure incorporates costs related to sustaining production. Copper and silver credits represent the expected revenue from the sale of these metals.

Sustaining capital expenditures and *Non-sustaining capital expenditures* are non-GAAP financial measures. Sustaining capital expenditures are defined as those expenditures required to sustain current operations and exclude all expenditures incurred at new operations or major projects at existing operations where these projects will materially benefit the operation. Non-sustaining capital expenditures are primarily costs incurred at 'new operations' and costs related to 'major projects at existing operations' where these projects will materially benefit the operation. A material benefit to an existing operation is considered to be at least a 10% increase in annual or life of mine production, net present value, or reserves compared to the remaining life of mine of the operation. Management uses the distinction of the sustaining and non-sustaining capital expenditures as an input into the calculation of all-in sustaining costs per ounce and all-in costs per ounce.

Management believes that the use of these non-GAAP measures will assist analysts, investors, and other stakeholders of the Company in understanding the costs associated with producing gold, understanding the economics of gold mining, assessing our operating performance, our ability to generate free cash flow from current operations and for planning and forecasting of future periods. However, the measures do have limitations as analytical tools as they may be influenced by the point in the lifecycle of a specific mine and the level of additional exploration or expenditures a company has to make to fully develop its properties. Accordingly, these non-GAAP measures should not be considered in isolation, or as a substitute for, analysis of our results as reported under GAAP. See "Non-GAAP and Other Financial Measures" on pages 47 to 54 in the Centerra Gold Management Discussion & Analysis, filed on *SEDAR+ (www.sedarplus.ca) or on EDGAR (www.sec.gov/edgar),* for the years ended December 31, 2025 and 2024 for a discussion of non-GAAP measures used in this document.

CERTIFICATE OF QUALIFIED PERSON CHRISTOPHER RICHINGS

I, Christopher Richings state that:

I am the Vice President, Technical Services at:

Centerra Gold Inc.
1 University Avenue, Suite 1800
Toronto, ON M5J 2P1

This certificate applies to the technical report titled Technical Report on the Kemess Project, North-Central British Columbia with an effective date of December 31, 2025 (the "Technical Report").

I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows: B.S. Mining Engineering from Colorado School of Mines (Golden, CO, USA 2002); registered Professional Engineer (P.Eng.) with Professional Engineers Ontario (PEO) and Engineers and Geoscientists British Columbia (EGBC); and experience in mine design, mine scheduling, mineral reserve estimation, mine operations, cost estimation and mine valuation for open pit, underground and precious and base metal projects in North and South America.

My most recent personal inspection of the property, that is the subject of the Technical Report occurred during July 8 to July 10, 2025, and was for a duration of three (3) days.

I am responsible for Items 2-5, 14-16, 19, 20, 22-24 and parts of Items 1, 12, 21, 25, 26 of the Technical Report.

I am not independent of the issuer as described in Section 1.5 of the Instrument.

I have had no prior involvement with this property prior to this Technical Report or employment with Centerra Gold Inc.

I have read the Instrument and the Items of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.

At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Items of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario this 4th day of March 2026.

(signed and sealed) *Christopher Richings*

Christopher Richings
P.Eng., PEO License # 100675583, EGBC License #58116

CERTIFICATE OF QUALIFIED PERSON CHEYENNE SICA

I, Cheyenne Sica state that:

I am the Director, Exploration Canada at:

Thompson Creek Metals Company, a subsidiary of Centerra Gold Inc.
299 Victoria Street
Prince George, BC V2L 5B8

This certificate applies to the technical report titled Technical Report on the Kemess Project, North-Central British Columbia with an effective date of December 31, 2025 (the "Technical Report").

I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows: I graduated from Queen's University with a Bachelor of Science Honours, Geological Sciences in 2010 and from the University of Toronto with a Master of Science, Earth Sciences in 2016. My relevant experience after graduation includes working for over fifteen years in exploration and geology for exploration and mine operating companies. I am registered as a P.Geo member in good standing with the Professional Geoscientists of Ontario (License #3005) since October 22, 2018, and with the Engineers & Geoscientists British Columbia (License #53969) since August 24, 2021.

My most recent personal inspection of the property that is the subject of the Technical Report occurred during July 8 to July 10, 2025 and was for a duration of 3 days.

I am responsible for Items 6-11, and parts of Items 12 and 26 of the Technical Report.

I am not independent of the issuer as described in Section 1.5 of the Instrument.

My prior involvement with the property that is the subject of the Technical Report is as follows. I have been involved with exploration on the property since Centerra acquired the property in 2018, with my first site visit to the property having occurred in 2019.

I have read the Instrument and the Items of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.

At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Items of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario this 4th day of March 2026.

(signed and sealed) *Cheyenne Sica*

Cheyenne Sica
P.Geo., PEO License #3005, EGBC License #53969

CERTIFICATE OF QUALIFIED PERSON GERARD ROWE

I, Gerard Rowe state that:

I am the Director, Engineering at:

Centerra Gold Inc.
1 University Avenue, Suite 1800
Toronto, ON M5J 2P1

This certificate applies to the technical report titled Technical Report on the Kemess Project, North-Central British Columbia with an effective date of December 31, 2025 (the "Technical Report").

I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows: I graduated from Dalhousie University in 2006 with a Bachelor of Engineering in Materials Engineering. I have 20 years' experience in the mining industry where I managed processing plants and specialize in the development of brownfield and green field projects. I am a registered as a Chartered Engineer with Engineers Ireland (License #465372) and a registered QP member of Mining and Metallurgical Society of America (License #1619)

My most recent personal inspection of the property that is the subject of the Technical Report occurred during August 19 – 21, 2025 and was for a duration of 3 days.

I am responsible for Items 13, 17 and 18, and parts of Items 1, 12, 21, 25 and 26 of the Technical Report.

I am not independent of the issuer as described in Section 1.5 of the Instrument.

My prior involvement with the property that is the subject of the Technical Report is as follows. I worked at the Kemess Mine from 2008 to 2010, during its operation under Northgate Minerals, where I held the positions of Mill Metallurgist and Shift Foreman. In 2025, I assumed the role of Project Director for Centerra Gold, overseeing all activities associated with the Kemess restart project as outlined in this Technical Report.

I have read the Instrument and the Items of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.

At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Items of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario this 4th day of March 2026.

(signed) *Gerard Rowe*

Gerard Rowe
C.Eng., License # 465372
MMSA QP, License # 1619

TABLE OF CONTENTS

1 Summary 1-1
1.1 Introduction 1-1
1.2 Location and Access 1-1
1.3 Geology and Exploration 1-2
1.4 Mineral Resources 1-6
1.5 Mineral Reserves 1-9
1.6 Mining 1-9
1.7 Metallurgy and Mineral Processing 1-11
1.8 Project Infrastructure 1-15
1.9 Environment, Community and Permitting 1-19
1.10 Markets and Contracts 1-21
1.11 Cost Estimates 1-22
1.12 Economic Evaluation 1-28
1.13 Taxation 1-29
1.14 Life-of-Mine Cash Flow Forecast 1-30
1.15 Study Conclusions 1-30
1.16 Recommendations 1-34

2 Introduction 2-1
2.1 Sources of Information 2-1
2.2 Contributing Persons and Site Inspections 2-2
2.3 Units of Measure 2-2

3 Reliance on Other Experts 3-1

4 Property Description and Location 4-1
4.1 Property Location 4-1
4.2 Property Description 4-6

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography 5-1
5.1 Access 5-1
5.2 Climate 5-2
5.3 Local Resources 5-2
5.4 Regional Infrastructure 5-2
5.5 Physiography 5-3

6 History 6-1
6.1 Property Ownership 6-1
6.2 Historical Exploration and Development Activities 6-3
6.3 Historical Production 6-6

7 Geological Setting and Mineralization 7-1

7.1 Regional Geology 7-1
7.2 Local and Property Geology 7-2
7.3 Lithology 7-4
7.4 Structure 7-7
7.5 Alteration 7-10
7.6 Mineralization 7-12

8 Deposit Types 8-1

9 Exploration 9-1
9.1 Recent Exploration Activities (2019–2024) 9-1
9.2 Summary and Interpretation of Exploration Activities 9-5

10 Drilling 10-1
10.1 Data Collection 10-4
10.2 Summary and Interpretation of 2019–2024 Drilling 10-11
10.3 Survey Control 10-14
10.4 Downhole Surveys 10-15
10.5 Core Recovery 10-17

11 Sample Preparation, Analyses, and Security 11-1
11.1 Pre-2019 Samples – Kemess Main ZONE 11-1
11.2 2019–2024 Centerra Samples – Kemess Main ZONE 11-2
11.3 Pre-2019 Samples – Kemess South 11-9

12 Data Verification 12-1
12.1 Geological Data 12-1
12.2 Geotechnical Data 12-5
12.3 Metallurgical Data 12-6
12.4 Environmental Data 12-7

13 Mineral Processing and Metallurgical Testing 13-1
13.1 Introduction 13-1
13.2 Testwork Overview 13-1
13.3 Historical Operating Data 13-20
13.4 Deleterious Elements 13-22
13.5 Leach Testwork 13-24
13.6 Recovery Estimates 13-25

14 Mineral Resource Estimates 14-1
14.1 Introduction 14-1
14.2 Methodology 14-1
14.3 Drillhole Database 14-2
14.4 Geological Interpretation and Modelling 14-4
14.5 Mineralization Controls and Grade Domain Modelling 14-9
14.6 Compositing Samples and Capping Assays 14-20

14.7 Specific Gravity 14-33
14.8 Variography 14-39
14.9 Block Model 14-43
14.10 Estimation Parameters 14-44
14.11 Block Model Validation 14-49
14.12 Classification 14-53
14.13 Mineral Resource Statement 14-55
14.14 Grade Sensitivity Analysis 14-59
14.15 Comparison to Previous Mineral Resource Statement 14-62

15 Mineral Reserve Estimates 15-1

16 Mining Methods 16-1
16.1 Introduction 16-1
16.2 Geotechnical Investigations 16-2
16.3 Open Pit Mine Design and Plan 16-21
16.4 Underground Mine Design and Plan 16-35
16.5 Mining Schedule 16-56
16.6 Mining Personnel 16-59
16.7 Open Pit and Underground Interaction 16-60

17 Recovery Methods 17-1
17.1 Summary 17-1
17.2 Plant Design 17-2
17.3 Process plant Description 17-4
17.4 Leach Plant and ADR 17-9
17.5 Energy, Water and Process Materials 17-14

18 Project Infrastructure 18-1
18.1 Off-site Road and Logistics 18-1
18.2 Site Infrastructure 18-1
18.3 Process Plant 18-4
18.4 Tailings Storage 18-4
18.5 Tailings Pipeline 18-28
18.6 Water Management 18-28

19 Market Studies and Contracts 19-1
19.1 Marketability 19-1
19.2 Metal Sales 19-1
19.3 Kemess Silver Stream 19-2
19.4 Contracts material to the project 19-2

20 Environmental Studies, Permitting, and Social or Community Impact 20-1
20.1 Environmental Studies 20-1
20.2 Tailings and Waste Rock Deposition Management, Monitoring and Water Management 20-6

20.3 Project Permitting 20-12
20.4 Social or Community Requirements 20-15
20.5 Closure Program 20-17

21 Capital and Operating Costs 21-1
21.1 Overview 21-1
21.2 Basis of Estimate 21-1
21.3 Capital Cost Estimates 21-2
21.4 Operating Cost Estimates 21-5

22 Economic Analysis 22-1
22.1 Assumptions 22-1
22.2 Taxation 22-2
22.3 Life-of-Mine Cash Flow Forecast 22-3
22.4 Sensitivity Analysis 22-5

23 Adjacent Properties 23-1

24 Other Relevant Data and Information 24-1

25 Interpretation and Conclusions 25-1
25.1 Geological Understanding 25-1
25.2 Mining 25-2
25.3 Metallurgy and Mineral Processing 25-3
25.4 Infrastructure Conclusions 25-4
25.5 Environmental and Social 25-5

26 Recommendations 26-1
26.1 Geology and Exploration 26-1
26.2 Geotechnical Engineering, Hydrogeology 26-2
26.3 Mining 26-3
26.4 Metallurgy and Mineral Processing 26-5
26.5 Infrastructure 26-5
26.6 Environmental and Social 26-7

27 References 27-1

28 Glossary of Units, Abbreviations, and Symbols 28-1
28.1 Glossary of Units 28-1
28.2 Glossary of Abbreviations 28-3
28.3 Glossary of Symbols 28-6

Tables

Table 1-1: Updated Kemess Resource Summary[1,4,5], as of December 31, 2025 1-8
Table 1-2: Mineral Resources in the Mine Plan 1-10
Table 1-3: KUG rock domains 1-12
Table 1-4: LOM capital and operating costs 1-22
Table 1-5: LOM capital costs[1] summary 1-23
Table 1-6: LOM mining production schedule 1-24
Table 1-7: Processing, concentrate and metals production schedule 1-24
Table 1-8: LOM operating costs 1-25
Table 1-9: LOM open pit mining costs 1-26
Table 1-10: LOM underground mining costs 1-26
Table 2-1: QPs and responsibilities 2-2
Table 6-1: Kemess historical ownership, exploration, and development timeline 6-1
Table 6-2: Historical production of Kemess South Mine 6-6
Table 10-1: Drilling programs summarized by year 10-3
Table 10-2: Drilling programs summarized by zone 10-3
Table 10-3: Transformation coordinates used for Kemess Mine Grid to UTM conversion 10-9
Table 11-1: Quantity of QAQC samples submitted for analysis by year 11-4
Table 11-2: CRMs selected for 2019-2024 Kemess drilling programs 11-5
Table 13-1: Historical testwork summary 13-1
Table 13-2: KUG rock domains 13-2
Table 13-3: Head assay summary (Kemess Underground) 13-5
Table 13-4: Summary statistics of SG tests on mineralized material 13-9
Table 13-5: Summary statistics of the SMC DWI 13-10
Table 13-6: Summary statistics of the SMC Axb 13-10
Table 13-7: Summary statistics of the Bond Ball Mill Work Index 13-11
Table 13-8: Summary of Abrasion Index tests results 13-11
Table 13-9: Flotation tests results (KM2911) 13-13
Table 13-10: Head grade characterization summary 13-16
Table 13-11: Copper sulphide mineral particle passing (KM7413) 13-17
Table 13-12: Comminution parameters (testwork average values) 13-18
Table 13-13: Comminution parameters (Main Zone vs Underground, average values) 13-22
Table 13-14: Summary of selected deleterious elements – Feed assays 13-23
Table 13-15: Summary of selected deleterious elements – Concentrate assays 13-23
Table 13-16: Locked cycle flotation composition and recoveries of first cleaner tails 13-24
Table 13-17: Cyanide leach tests results summary 13-25
Table 13-18: Main Zone locked cycle test results summary (2025 program) 13-26
Table 13-19: Locked cycle test summary (SGS 2019) 13-26
Table 14-1: Modelled lithology units 14-4
Table 14-2: List of major modelled faults 14-5
Table 14-3: Modelled alteration assemblages 14-6
Table 14-4: Modelled lithology units 14-8
Table 14-5: List of major modelled faults at Kemess South 14-9
Table 14-6: Kemess Main Zone grade domain modelling criteria 14-10
Table 14-7: Kemess South grade domain modelling criteria 14-16
Table 14-8: Summary statistics of length-weighted gold assays 14-21
Table 14-9: Summary statistics of length-weighted copper assays 14-22
Table 14-10: Summary statistics of length-weighted silver assays 14-22

Table 14-11: Summary statistics of length-weighted, capped vs uncapped 2 m gold composites 14-24
Table 14-12: Summary statistics of length-weighted, capped vs uncapped 2m copper composites 14-25
Table 14-13: Summary statistics of length-weighted, capped vs uncapped 2m silver composites 14-26
Table 14-14: Summary statistics of length-weighted gold assays 14-28
Table 14-15: Summary statistics of length-weighted copper assays 14-28
Table 14-16: Summary statistics of length-weighted silver assays 14-28
Table 14-17: Summary statistics of length-weighted, capped vs uncapped 2 m gold composites 14-30
Table 14-18: Summary statistics of length-weighted, capped vs uncapped 2 m copper composites 14-31
Table 14-19: Summary statistics of length-weighted, capped vs uncapped 2 m silver composites 14-32
Table 14-20: SG measurement methodology in the Kemess District Project 14-33
Table 14-21: Statistics of SG measurements used in the Kemess Main Zone MRE 14-34
Table 14-22: Statistics of 2 m SG composites – capped vs uncapped 14-35
Table 14-23: Statistics of SG measurements used in the Kemess South MRE 14-37
Table 14-24: Statistics of 2 m SG composites – capped vs uncapped 14-38
Table 14-25: Summary of variogram parameters used in the Kemess Main Zone MRE 14-42
Table 14-26: Summary of variogram parameters used in the Kemess South MRE 14-42
Table 14-27: Leapfrog Edge block model definition 14-43
Table 14-28: Data and search parameters for Kemess Main Zone estimation 14-46
Table 14-29: Data and search parameters for Kemess South estimation 14-47
Table 14-30: Gold value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates 14-51
Table 14-31: Copper value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates 14-51
Table 14-32: Silver value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates 14-51
Table 14-33: Parameters for classification 14-54
Table 14-34: Optimization parameters used to generate conceptual Kemess pits and underground mineable shapes 14-56
Table 14-35: Updated Kemess Resource Summary(1,4,5), as of December 31, 2025 14-58
Table 14-36: Kemess Main open pit Indicated and Inferred Mineral Resource sensitivity; base case scenario NSR cut-off CA$15.97/t 14-59
Table 14-37: KUG Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off CA$54.10/t 14-60
Table 14-38: Kemess East Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off CA$54.10/t 14-61
Table 14-39: Kemess South Open Pit Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off of CA$13.27/t 14-62
Table 14-40: Mineral Resources (December 31, 2025) comparison to April 15, 2025 14-63
Table 16-1: Basis of mineral inventory for PEA Mine Plan 16-1
Table 16-2: PEA Mine Plan mineral inventory 16-1
Table 16-3: Laboratory UCS testing summary 16-7
Table 16-4: Laboratory shear testing summary 16-7
Table 16-5: Summary of rock mass characteristics 16-9
Table 16-6: Summary of rock mass permeabilities 16-10
Table 16-7: Recommended pit slope configurations 16-11
Table 16-8: Calculated maximum strike sidewall lengths in all ground conditions at various mining depths 16-20
Table 16-9: Open pit block model prototype information 16-22
Table 16-10: Block model field definitions 16-22
Table 16-11: Main Zone estimate of mineral recovery 16-23
Table 16-12: Pit shell optimization inputs 16-23
Table 16-13: Pit slope angles by pit wall sector 16-24
Table 16-14: Mining physicals by pit operations phase 16-26
Table 16-15: Open Pit Mining Schedule inputs 16-30
Table 16-16: Annual tonnes mined by pit phase 16-32
Table 16-17: Open pit mine production 16-32
Table 16-18: Open pit mining equipment list 16-34
Table 16-19: Open pit conveyor drive schedule 16-35

Table 16-20: Cost included in mine cut-off grade 16-37
Table 16-21: Preliminary estimates of the various cut-off values 16-37
Table 16-22: Dilution and recovery summary for KUG Project 16-38
Table 16-23: Preliminary MSO parameters 16-39
Table 16-24: Key economic screening parameters 16-39
Table 16-25: Stope dimensions by mining method 16-42
Table 16-26: Key orphan analysis parameters 16-42
Table 16-27: Development activity and equipment rates 16-43
Table 16-28: Longhole production task rates and durations 16-43
Table 16-29: Total and annual lateral development schedule for KUG 16-44
Table 16-30: Total and Annual Vertical Development Schedule for Kemess Underground 16-44
Table 16-31: Total production, by resource classification, for KUG 16-46
Table 16-32: Total and annual mine mineralized material schedule for KUG 16-47
Table 16-33: Total underground mining quantities 16-47
Table 16-34: Backfill binder quantities and curing durations 16-50
Table 16-35: Ventilation demand estimate for the KUG mine 16-52
Table 16-36: Level ventilation demand estimate for KUG mine 16-52
Table 16-37: Kemess power estimate (mining activities only) 16-56
Table 16-38: LOM open pit and underground ore delivery to Process Plant 16-58
Table 17-1: Kemess process design criteria 17-3
Table 17-2: Reagents List 17-13
Table 17-3: Leach circuit reagents 17-13
Table 17-4: Annual reagent and consumables requirements 17-14
Table 18-1: Seismic criteria and 2020 NBCC PGA values 18-10
Table 18-2: LOM annual tailings production 18-11
Table 18-3: IDF and wave run-up minimum dam freeboard 18-13
Table 18-4: KUG TSF tailings management strategy 18-14
Table 18-5: KS TSF tailings management strategy 18-15
Table 18-6: Channel/Conduit design parameters 18-20
Table 18-7: KS TSF dam raise sequencing 18-21
Table 18-8: Minimum factors of safety for dam stability 18-22
Table 18-9: Material strength parameters used in stability analysis 18-25
Table 18-10: KS TSF stability analyses results 18-26
Table 18-11: Localized failure Stability Analyses results 18-27
Table 18-12: Diversion dams and North dam Stability Analyses results 18-27
Table 21-1: LOM capital and operating costs 21-1
Table 21-2: LOM capital costs[1] summary 21-2
Table 21-3: LOM operating costs 21-5
Table 21-4: LOM mining production schedule 21-7
Table 21-5: LOM open pit mining costs 21-8
Table 21-6: LOM underground mining costs 21-8
Table 21-7: LOM processing plant production schedule 21-10
Table 22-1: Cash flow summary (in millions of US dollars, unless otherwise indicated) 22-4
Table 22-2: After-tax $NPV_{5\%}$ sensitivities to changes in assumptions 22-5

Figures

Figure 1-1: Kemess location and access road 1-2
Figure 1-2: Kemess Property exploration drilling 1-5
Figure 1-3: LOM production summary 1-11
Figure 1-4: Kemess South TSF model showing containment dams 1-17
Figure 4-1: Kemess Project location 4-2
Figure 4-2: Kemess mineral claims 4-3
Figure 4-3: Mineral tenure information as recorded online with BC Mineral Titles 4-4
Figure 4-4: Aerial view of Kemess Project site with project extent overlay 4-6
Figure 5-1: Road access to Kemess Project 5-1
Figure 7-1: Geological map of Toodoggone mineral district 7-1
Figure 7-2: Geological map of the Kemess deposits 7-3
Figure 7-3: Geological map of the Kemess North deposits 7-9
Figure 7-4: Geological map of the Kemess South deposits 7-9
Figure 9-1: Kemess exploration targets 9-2
Figure 9-2: Kemess gold soil geochemistry 9-3
Figure 9-3: Kemess North Trend brownfield exploration targets 9-5
Figure 9-4: Schematic illustration of the Kemess Main Zone porphyry 9-7
Figure 10-1: Drill hole collar map Kemess Project 10-2
Figure 10-2: Drill hole collar map Kemess North 10-6
Figure 10-3: Drill hole collar map Kemess South 10-10
Figure 10-4: Geological and gold mineralization cross-section at Kemess East 10-13
Figure 10-5: Histograms plotting total core recovery (%) for Kemess Main Zone from historical pre-2019 drilling 10-18
Figure 10-6: Histograms plotting total core recovery (%) for Kemess Main Zone from Centerra drilling, 2019–2024 10-18
Figure 10-7: Histograms plotting total core recovery (%) for Kemess South (2004–2007) drilling and Centerra 2019 drilling 10-19
Figure 11-1: Blanks for gold fire assay (FA430) from 2019–2024 11-6
Figure 11-2: Coarse reject duplicates for copper (MA200) from 2017–2025 11-7
Figure 11-3: Standard OREAS 152b by sequence for copper (MA200) 11-8
Figure 11-4: Standard OREAS 520 by sequence for gold (MA200) 11-8
Figure 11-5: Blanks for gold fire assay (AA23) from 2003–2005 11-11
Figure 11-6: Coarse reject duplicates for gold (AA23 – ppm) and copper (AA49 – %) from 2003–2005 11-12
Figure 11-7: Standard OxE-21 by sequence for gold (AA23) 11-13
Figure 11-8: Standard OxE-29 by sequence for gold (AA23) 11-13
Figure 13-1: Sample locations for comminution testwork (KM3442, 2012) 13-3
Figure 13-2: Sample locations for comminution testwork (KM4730, 2015) 13-4
Figure 13-3: Overall copper sulphide liberation 13-6
Figure 13-4: Sulphide mineralogy (ALS-KM4379) 13-6
Figure 13-5: Gold mineralogy size distribution chart (SGS Report: 17102-01) 13-8
Figure 13-6: Gold mineral liberation (%) (SGS Report: 17102-01) 13-8
Figure 13-7: Mass recovery vs grind size 13-12
Figure 13-8: Copper recovery – 2019 KUG testwork 13-13
Figure 13-9: Gold recovery – 2019 KUG testwork 13-14
Figure 13-10: Blend composite grade recovery curves 13-14
Figure 13-11: Overall copper sulphide liberation (KM7413) 13-17
Figure 13-12: Gold recovery – 2025 Main Zone testwork 13-20
Figure 13-13: Copper recovery – 2025 Main Zone testwork 13-20
Figure 13-14: Plant throughput – Historical plant data 13-21

Figure 13-15: Metals head grades – Historical plant data 13-21
Figure 13-16: Metals recovery – Historical plant data 13-22
Figure 13-17: Gold leach kinetics of samples from copper first cleaner tails 13-25
Figure 14-1: Kemess property drillhole collar locations 14-3
Figure 14-2: Litho-structural model for Kemess deposits shown in long-section view, looking north-northwest 14-5
Figure 14-3: Kemess Main Zone Trend fault model 14-6
Figure 14-4: Alteration model for Kemess Main Zone Trend deposits (long-section view, looking north-northwest) 14-7
Figure 14-5: Long section of the Kemess deposits showing leach and broken zones, looking north-northwest 14-7
Figure 14-6: Kemess South litho-structural and supergene enrichment models (cross-section view) 14-8
Figure 14-7: Kemess South fault model, viewed obliquely from above towards the northeast 14-9
Figure 14-8: Log probability plot of all gold assays in the KUG area 14-11
Figure 14-9: Log probability plot of all gold assays in the Kemess East area 14-12
Figure 14-10: Modelled gold domains at Kemess Main Zone 14-13
Figure 14-11: Long-section views of the modelled copper domains at Kemess 14-14
Figure 14-12: Long-section view of the modelled silver domains at Kemess 14-15
Figure 14-13: Log probability plot of all gold assays in Kemess South 14-17
Figure 14-14: Oblique view of Kemess South mineralized gold domains 14-18
Figure 14-15: Oblique view of the Kemess South mineralized copper domains 14-19
Figure 14-16: Oblique view of the Kemess south mineralized silver domains 14-20
Figure 14-17: Sample length distribution and associated statistics 14-21
Figure 14-18: Examples of grade capping treatment based on log probability curve 14-23
Figure 14-19: Sample length distribution and associated statistics 14-27
Figure 14-20: Composite data distribution used for the determination of capping values of the main gold domain of Kemess South (within Fault Block 3) 14-29
Figure 14-21: Long section view showing the distribution of SG measurements in the Kemess Main Zone deposit 14-33
Figure 14-22: Log probability plot of 2 m SG composites (a value of 3.2 was used to cap composites) 14-36
Figure 14-23: Oblique view showing the distribution of SG measurements in the Kemess South deposit 14-37
Figure 14-24: Log probability plot of 2 m SG composites at Kemess South (a value of 3 was used to cap composites) 14-39
Figure 14-25: Experimental variogram for the Nugget_KUG_0.2 gold domain at Kemess Main Zone 14-40
Figure 14-26: Experimental variogram for the copper domain in Fault Block 3 at Kemess South 14-41
Figure 14-27: Spatial setting of Kemess block models; KUG and Kemess East 5x5x5 m block centroids match those of the larger Kemess open pit 15x15x15 m block model 14-44
Figure 14-28: Swath plot validation of KUG low grade domain displaying both hard and soft boundary capped gold composites against block model parallel estimates 14-45
Figure 14-29: East-west cross section through the Kemess Main open pit conceptual pit shell and KUG MSOs 14-48
Figure 14-30: East-west cross section through the Kemess South block model area showing good agreement between 2 m composites and block grades 14-49
Figure 14-31: Cross strike gold swath plots through the Kemess Main Zone deposits 14-52
Figure 14-32: Cross strike gold swath plot of the mineralized gold domain at Kemess South 14-53
Figure 14-33: Oblique view of Kemess Main Zone 15x15x15 classified block model; pre and post manual smoothing 14-55
Figure 14-34: Oblique view of Kemess South 15x15x15 classified block model; pre and post manual smoothing 14-55
Figure 16-1: Isometric view of the integrated open pit and underground concept for the Kemess PEA 16-2
Figure 16-2: Geotechnical drillhole locations 16-6
Figure 16-3: Precedent for hard rock slopes 16-13
Figure 16-4: Major structures affecting underground design 16-15
Figure 16-5: RQD% iso-surfaces in the Mine Plan 16-17
Figure 16-6: KUG stope designs and depths plotted on the Modified Stability Graph 16-19
Figure 16-7: Typical drift cross-sections showing rock support designs 16-20
Figure 16-8: Paste backfill design criteria 16-21
Figure 16-9: RF plot 16-25
Figure 16-10: Phase 0 (Ridgeline Crossing) 16-27

Figure 16-11: Phase 1 Mine Plan 16-27
Figure 16-12: Phase 2 Mine Plan 16-28
Figure 16-13: Phase 3 Mine Plan 16-28
Figure 16-14: Main Zone Pit and PAG facility layout 16-29
Figure 16-15: Main Zone Pit Phase Mining Schedule 16-31
Figure 16-16: Open pit mined tonnes and metal grades 16-33
Figure 16-17: Gold and copper recovery formulae 16-37
Figure 16-18: KUG typical development level layout 16-40
Figure16-19: Isometric view of the proposed KUG mine 16-41
Figure 16-20: Annual lateral development for KUG 16-44
Figure 16-21: Main haulage pathways at KUG mine 16-45
Figure 16-22: Mined mineralisation for KUG mine 16-46
Figure 16-23: Schematic of paste backfill activity and associated infrastructure 16-50
Figure 16-24: Ventilation circuit at KUG mine 16-51
Figure 16-25: Secondary egress for Kemess Mine 16-53
Figure 16-26: Underground workshop at KUG mine 16-54
Figure 16-27: Crushers and conveyors at KUG mine 16-55
Figure 16-28: Annual combined open pit and underground production and metal grades 16-57
Figure 16-29: Process feed summary 16-59
Figure 16-30: On-site personnel 16-59
Figure 17-1: Kemess overall process flow diagram 17-5
Figure 17-2: Underground and open pit crusher and conveyor routing 17-6
Figure 17-3: Leach plant process flow diagram 17-10
Figure 18-1: Typical section through KS TSF main dam 18-5
Figure 18-2: Detailed section of KS TSF main dam 18-5
Figure 18-3: Modelled KS TSF 18-8
Figure 18-4: KUG TSF filling curve 18-14
Figure 18-5: KS TSF filling curve 18-15
Figure 18-6: KS TSF main dam cross-section 18-23
Figure 18-7: North saddle dam cross-section 18-23
Figure 18-8: East diversion dam cross-section 18-23
Figure 18-9: North diversion dam cross-section 18-24
Figure 18-10: South diversion dam cross-section 18-24
Figure 18-11: Simplified section for analysis 18-26
Figure 20-1: Conceptual model for water management at Kemess 20-10
Figure 20-2: Waste rock storage (gray) and Kemess property limit (yellow line) 20-12

1 SUMMARY

1.1 INTRODUCTION

Centerra Gold Inc. (Centerra), a global mining company organized under the laws of Canada, is engaged in the acquisition, exploration, development, and operation of mineral properties. In 2018, Centerra acquired its 100% interest in AuRico Metals Inc. (AuRico), which owned a number of assets including the idle Kemess Mine. References to “Centerra” herein shall include its affiliates.

Royal Oak Mines developed the Kemess South operation in 1998, and Northgate Minerals operated the mine from 2001 until 2011, when the mine was closed. Over its 13-year operation, Kemess processed 218 Mt of ore producing gold-bearing copper concentrates containing 2.98 Moz gold and 360.5 kt copper.

This Technical Report summarizes a Preliminary Economic Assessment (PEA) completed in 2025 on the Kemess Restart Project (KRP or the Project), incorporating changes to the mineral inventory and Project Plan as compared to the prior Technical Report on Kemess, filed in 2016 by AuRico.

The Technical Report discloses an increase in Mineral Resources and the results of a PEA study to establish a new mining area and restart the existing process plant. Mineral Resources have been estimated according to CIM Definition Standard for Mineral Resources and Mineral Reserves (CIM, May 2014) and CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines (CIM, Nov 2019).

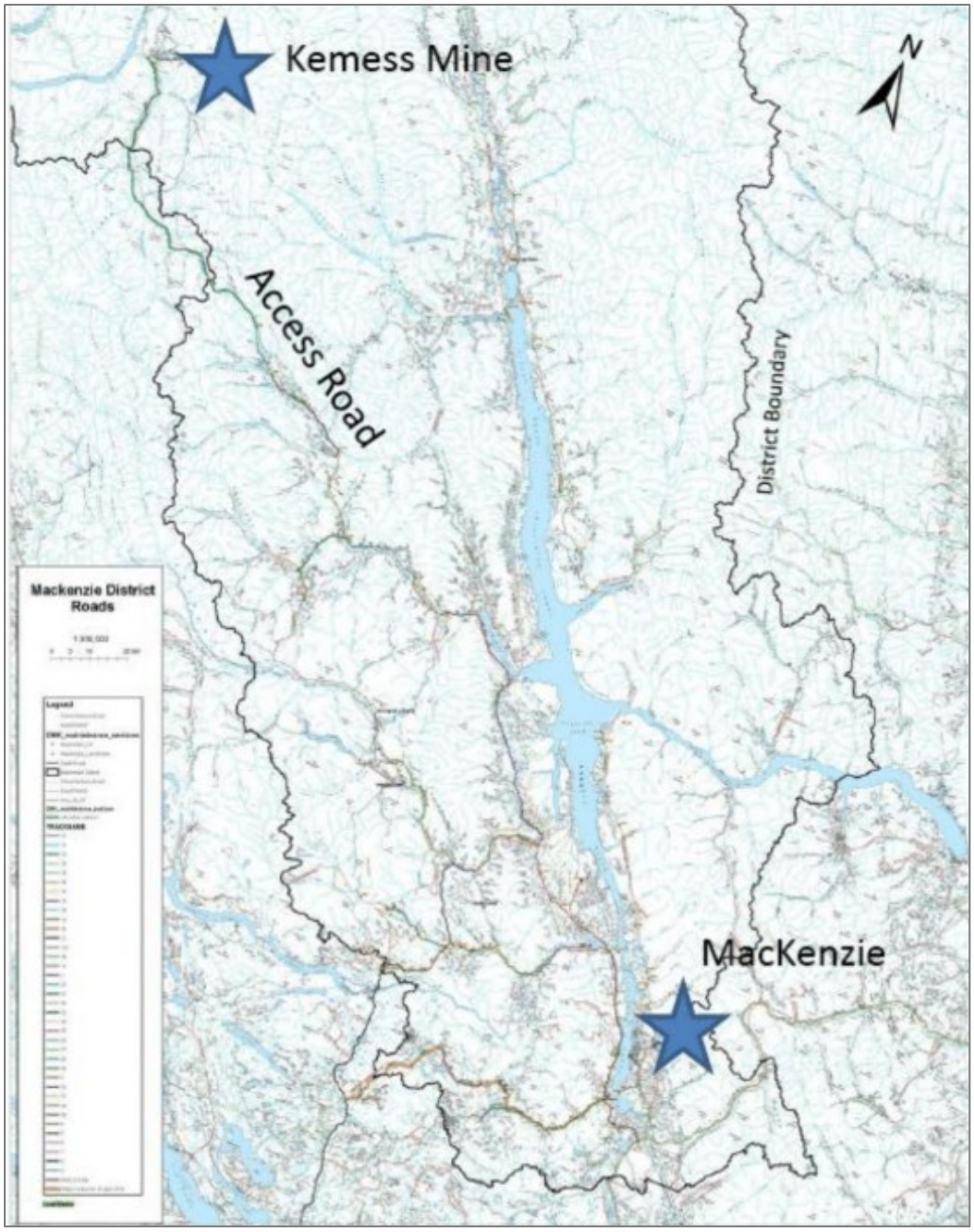
The report has been prepared pursuant to Canadian Securities Administrators’ NI 43-101 Standards for Disclosure for Mineral Projects (2011), Companion Policy 43- 101CP, and Form 43-101F1 Technical Report.

The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

1.2 LOCATION AND ACCESS

Kemess is located in remote north-central, British Columbia, Canada, approximately 325 km north-northwest of Fort St. James and 200 km northeast of Smithers. A 340 km access road connects Kemess to Mackenzie, which will be the supply centre for the mine. Figure 1-1 shows the location of Kemess relative to Mackenzie and the connecting access road. Mackenzie is serviced by paved highways, a railway, and an airport.

Figure 1-1: Kemess location and access road



1.3 GEOLOGY AND EXPLORATION

1.3.1 Geology and Mineralization

The Kemess Project is located at the northeastern margin of the Stikine Terrane of the Intermontane Belt within the prospective Toodoggone district in British Columbia. The Kemess area is made up of four main lithostratigraphic units including three volcano-sedimentary island arc assemblages referred to as the Asitka Group, the Takla Group, and the Hazelton Group.

Within the Kemess property there are several known porphyry copper-gold deposits including Kemess South, Nugget, Kemess Underground (KUG), and Kemess East, as well as several other mineral prospects and showings. The Black Lake suite comprises the plutonic rocks observed on the Kemess property, including syn-mineralization and post-mineralization intrusions. These Black Lake suite intrusive rocks are associated with most known mineralized porphyry systems in the Toodoggone district.

The Kemess KUG deposit lies approximately 6 km north of the existing Kemess South processing plant and other mine infrastructure. The KUG and Kemess East calc-alkaline porphyry deposits host copper-gold-silver and molybdenum mineralization. At KUG, the main sulphides hosting gold and copper mineralization are chalcopyrite and pyrite. Other accessory mineral phases include pyrrhotite and a low iron sulphate zone with gypsum/anhydrite and pyrite. At Kemess East, higher-grade copper-gold mineralization is characterized by strong secondary biotite and quartz alteration and lesser chlorite alteration in the plutonic rocks.

1.3.2 Exploration, Drilling, Sampling, Assays, Records

Since acquiring the Kemess Project, Centerra has conducted various exploration programs in 2019, 2020, and 2024, at both Kemess North and Kemess South, including geophysical surveys and diamond drilling campaigns.

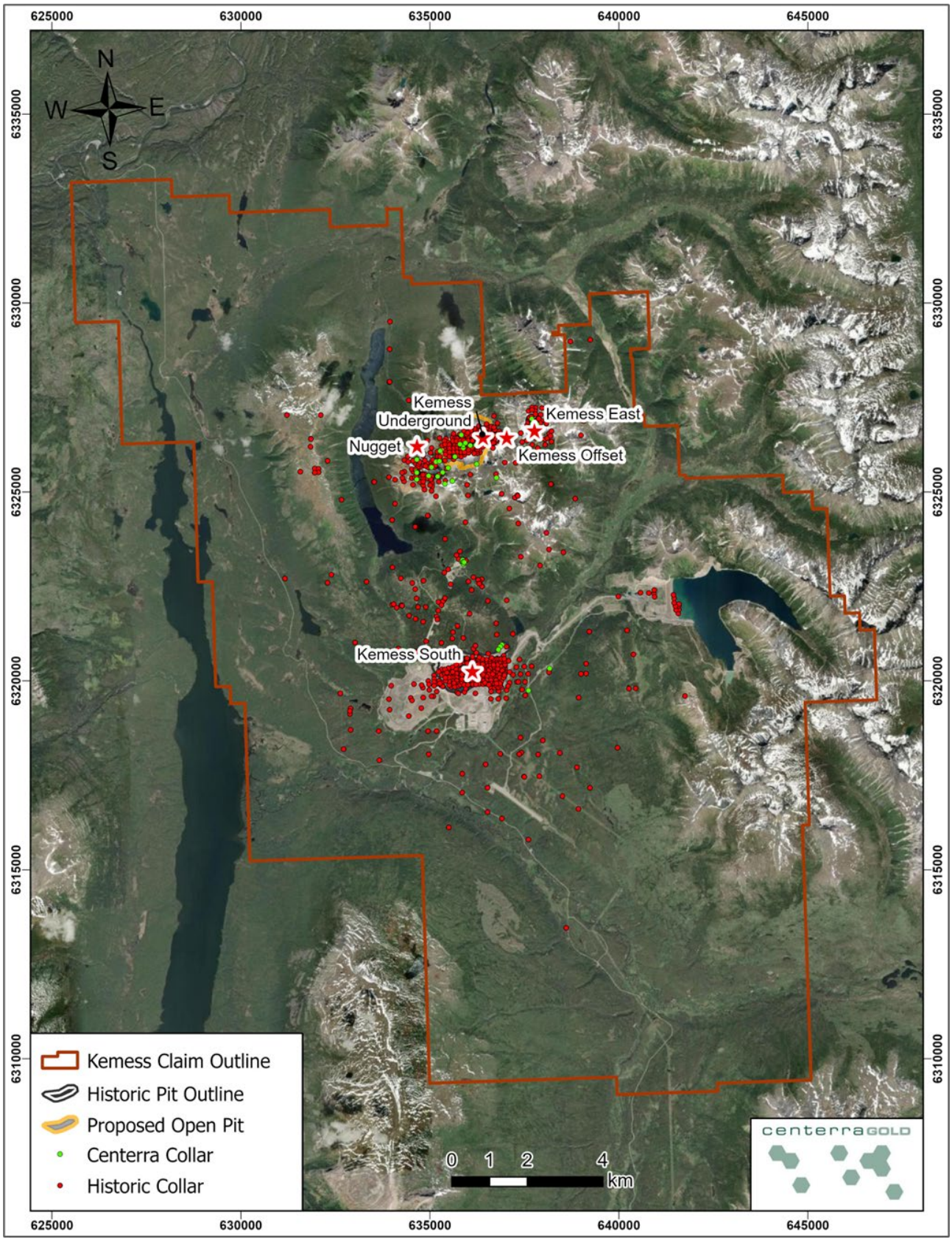
Diamond drilling at the Kemess property totals over 330,000 m of drill core in nearly 900 drill holes (Figure 1-2), and was designed for mineral exploration, resource delineation and infill, to obtain metallurgical samples, to condemn areas planned for infrastructure, and to gather geotechnical and environmental information. Since 2019, Centerra has completed 27,357 m of diamond drilling in 51 drill holes at Kemess North and Kemess South.

The 2019–2024 drilling by Centerra has continued to define the Kemess North mineralization trend, an east-westerly striking prospective belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east, the trend includes the Nugget, KUG, Kemess Offset, Kemess East, and Hilda South targets, with a general trend of deposits occurring at greater depths towards the east.

Diamond drill core samples were collected from predominantly NQ diameter diamond drill core, as well as HQ diameter (63.5 mm) drill core. Sample intervals were predominantly of 2 m core length. All core samples were marked with a unique sample ID number and sample intervals were identified by sample tags in the core box. Technicians logged the core for geotechnical characteristics, and the geologist logged the core in detail. Drill core was then marked with the sample intervals, cut lines, and assigned sample numbers. The wet and dry drill core was photographed prior to being sent for cutting. The NQ and HQ drill core was cut in half using a diamond blade electric core saw, with one half placed in a

sample bag for shipping to the laboratory and the other half placed back in the core box for future reference.

Figure 1-2: Kemess Property exploration drilling



From 2019 to present, drill samples were cut at the project site and half-core samples were sent to Bureau Veritas (BV) in Vancouver, British Columbia for sample preparation and pulverization. At BV, all drill core samples were analyzed for precious and base metals, as well as multi-elements.

A subset of 3.8% of samples from 2019–2024 drill core samples were sent to SGS Laboratories (SGS) in Vancouver, British Columbia, as an independent check for analytical bias and accuracy. At SGS, samples were analyzed for similar precious and base metals as completed by BV.

Further to the quality assurance/quality control (QA/QC) procedures implemented for the 2017–2025 drilling programs, routine data checks are performed to ensure the assays in the drill hole database are checked against assay certificates received by the lab. In 2025, the Centerra database management team developed a series of structured SQL queries to assess the integrity, completeness, and consistency of assay data within the acQuire database. Each query contributes to identifying potential data quality issues that could impact downstream geological modelling and resource estimation processes.

As part of the audit process conducted directly on the acQuire database, a set of QA/QC reports was prepared using the acQuire QA/QC Report object to evaluate the analytical reliability of the assay data. These reports represent a standalone quality assessment, independent of the export mechanisms, and are intended to provide additional confidence in the integrity of the underlying data.

All Kemess drill core is stored at the Kemess mine site.

In the opinion of the Qualified Person (QP) for this Item of the Technical Report, sample preparation, security, and analytical procedures utilized during Kemess North drilling programs were adequate and conducted according to CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The QP recommends re-assaying a selection of pre-2003 Kemess South historical drill core, if available, to verify the historical assay results.

1.4 MINERAL RESOURCES

The Mineral Resource estimate (MRE) is a reasonable representation of the global gold, silver and copper mineral resources of the Project at the current level of sampling. It has been completed in conformity with the widely accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines (CIM, Nov 2019) and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 Form F1. The resource includes four deposits – two open pits – Kemess Main and Kemess South, and two underground deposits – Kemess Underground (KUG), and Kemess East.

Routine validation of the drilling database is performed to maintain data accuracy and consistency. Historical drillhole and assay data used in the MRE have been reviewed for completeness, accuracy,

and consistency and potential biases were investigated. The data were deemed appropriate for inclusion in the MRE.

The modelled mineralization of Kemess Main Zone measures 3.9 km along strike, 0.6 km wide, and 1.7 km deep. Excluding KE, the Kemess Main Zone is 0.8 km deep. The MRE presented herein considers all current lithological and alteration data to support the statistical evaluation of gold, copper, and silver assays. The model incorporates exploration drillhole data up to March 24, 2025, and utilizes gold mineralization domains that are based on a 0.2 g/t Au grade, copper mineralization domains based on a 0.2% Cu grade, and silver mineralization domains based on 1.5 g/t Ag grade. Mineralization is largely hosted in the Black Lake lithological unit.

At Kemess South, the resource area measures 2.1 km along strike, 0.7 km across strike, and is 0.5 km deep. The model incorporates exploration drillhole data up to November 18, 2025, and utilizes gold mineralization domains that are based on a 0.2 g/t Au grade, copper mineralization domains based on a 0.2% Cu grade, and silver mineralization domains based on 1 g/t Ag grade. Like the Kemess Main Zone, mineralization is largely hosted in the Black Lake lithological unit.

The global Kemess database comprises 907 diamond drillholes (338,304 m), yielding 159,897 gold, 159,928 copper, and 136,311 silver assays.

The effective date of the Kemess Main Zone drilling database is March 24, 2025. The 2025 Kemess Main Zone MRE utilises 417 drillholes (247,763 m), excluding 468 drillholes (88,302 m) as they are outside of the main mineralized domains.

The effective date of the Kemess South drilling database is November 18, 2025. The 2025 Kemess South MRE utilises 273 drillholes (46,309 m), excluding 634 drillholes (291,995 m) as they are outside of the main mineralized domains.

The Kemess Main Zone database contains 52,722 specific gravity (SG) measurements of which 51,065 measurements were used in the resource estimate. The average SG is 2.75 from a range of 2.69 to 2.78. The SG database at Kemess South includes 3,511 SG measurements of which 3,161 SG measurements used for the resource estimate. The average SG of Kemess South rock is 2.67 from a range of 2.55 to 2.76.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves.

CIM Definition Standard for Mineral Resources and Mineral Reserves (CIM, May 2014) define a Mineral Resource 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."*

The "reasonable prospects for eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery. Centerra considers that the gold, copper, and silver mineralization of the Kemess Main Zone and Nugget deposits is amenable to open pit extraction, and that of the Kemess Underground and Kemess East deposits is amenable to underground extraction. Centerra used a pit optimizer to assist with determining which portions of the gold deposits show "reasonable prospect for eventual economic extraction" from an open pit and to assist with selecting reporting assumptions. The pit optimization assumptions used in their development are summarized in Table 14-34.

The Mineral Resource Statement for the Kemess deposits is presented in Table 1-1. The effective date of the Mineral Resource Statement is December 31, 2025.

Table 1-1: Updated Kemess Resource Summary[1,4,5], as of December 31, 2025

Indicated Mineral Resources[2]							
	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)	Copper Grade (%)	Contained Copper (Mlbs)	Silver Grade (g/t)	Contained Silver (koz)
Open Pit - Main Zone	170,513	0.30	1,668	0.15	575	1.12	6,155
Open Pit - Kemess South	13,204	0.37	158	0.13	38	0.68	289
Underground - Kemess UG	33,223	0.82	877	0.36	265	2.48	2,652
Underground - Kemess East	27,491	0.64	565	0.44	268	1.91	1,684
Total Indicated	**244,431**	**0.42**	**3,269**	**0.21**	**1,146**	**1.37**	**10,780**
Inferred Mineral Resources[3]							
	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)	Copper Grade (%)	Contained Copper (Mlbs)	Silver Grade (g/t)	Contained Silver (koz)
Open Pit - Main Zone	237,050	0.30	2,299	0.13	682	1.06	8,108
Open Pit - Kemess South	198	0.34	2	0.08	0	0.42	3
Underground - Kemess UG	20,094	0.74	481	0.33	148	2.22	1,433
Underground - Kemess East	42,252	0.57	772	0.42	393	1.92	2,602
Total Inferred	**299,593**	**0.37**	**3,555**	**0.19**	**1,223**	**1.26**	**12,146**

Notes:

1. Mineral resources are stated in accordance with CIM (2014) Definitions as incorporated by reference into NI 43-101. Mineral Resources are estimated and have an effective date of December 31, 2025.
2. Mineral resources do not have demonstrated economic viability.
3. Inferred mineral resources have a lower level of confidence as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the inferred mineral resources will ever be upgraded to a higher category.
4. Centerra's equity interests are as follows: Kemess Main, Kemess South, Kemess UG, Kemess East 100%.
5. Numbers may not add precisely due to rounding.

Additional Resource footnotes:

- A conversion factor of 31.1035 grams per troy ounce of gold is used in the mineral reserve and resource estimates.
- The mineral resources are reported based on a gold price of $2,400/oz, a copper price of $4.00/lb, a silver price of $25.00/oz and an exchange rate of 1USD:1.33CAD.
- The Kemess Main open pit mineral resources (including the Nugget zone) are constrained by a pit shell and are reported based on a net smelter return (NSR) cut-off of $12.01/t (CA$15.97/t) that considers materials handling costs, metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges to determine economic viability. A dilution factor of 0% and a mining recovery of 100% is used.
- The Kemess South open pit mineral resources are constrained by a pit shell and are reported based on a NSR cut-off of $9.98/t (CA$13.27/t) that considers metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges to determine economic viability. A dilution factor of 0% and a mining recovery of 100% is used.
- The Kemess Underground mineral resource is constrained by optimized stope shapes using commercially available software. Optimized stope shapes were included where the estimated average stope NSR exceeded a minimum stope cut-off value of $40.68/t (CA$54.10/t), representing the estimated breakeven value required to cover mining, processing, general and administrative, and sustaining capital costs. Economic screening was performed on stope shapes to ensure reasonable prospects for eventual economic extraction. Dilution was estimated using equivalent linear overbreak sloughing (“ELOS”) for each slope type and ore-waste contacts, which vary between zero and 1.25 m. Mining recovery of 93% was applied to all stopes.
- The Kemess East underground mineral resource is constrained by optimized stope shapes using commercially available software. Optimized stope shapes were included where the estimated average stope NSR exceeded a minimum stope cut-off value of $40.68/t (CA$54.10/t), representing the estimated breakeven value required to cover mining, processing, G&A, and sustaining capital costs. Economic screening was performed on stope shapes to ensure reasonable prospects for eventual economic extraction. Dilution was estimated using ELOS for each slope type and ore-waste contacts, which vary between zero and 1.25 m. Mining recovery of 93% was applied to all stopes.
- The Kemess Main open pit shell was restricted to a minimum floor elevation of 1,355 metres above sea level (“masl”) and the Kemess Underground optimized stope shapes were restricted to a maximum elevation of 1,355 masl, to represent the conceptual transition between open pit and underground mining zones for resource estimation purposes.
- A portion of the mineral resource estimate is included in the economic analysis for the PEA, which is limited to the Kemess Main open pit and Kemess Underground zones. This is a conservative subset that reflects mining, processing and economic assumptions. It is important to note that the PEA mining inventory is not a mineral reserve and does not demonstrate economic viability. The subset of the mineral resource used in the PEA was based on a gold price of $2,000/oz, a copper price of $3.75/lb, a silver price of $22.50/oz and an exchange rate of 1USD:1.33CAD.

The previous Technical Report for Kemess was produced by Golder Associates Ltd in 2017, which focussed on the KUG and Kemess East deposits. Compared to the 2016 MRE, the current, updated estimate reflects changes resulting from additional drilling, updated geological interpretation, revised estimation parameters, the application of updated economic assumptions, and a change in mining methods. The Kemess open pit incorporates mineralized material previously considered part of a block cave concept for Kemess Underground, which has increased the size and quality of the resources considered for open pit mining. The tonnages and mineral inventory in KUG have decreased while grades have increased due to the Kemess open pit mining out a portion of the previously contemplated block cave, as well as the change of mining method to longhole open stoping. The tonnages in Kemess East also decreased while grades increased due to the lower dilution of longhole open stoping compared with the previous block cave mining method.

1.5 MINERAL RESERVES

Mineral Reserves have not been declared as a result of the PEA.

1.6 MINING

A combination of open pit and underground mining methods is proposed for Kemess. The shallow, lower grade Main Zone deposit, which is located directly above KUG, is suited for open pit mining. A longhole open stoping method is proposed for the KUG deposit. The underground method considers the depth

of the high-grade core, the bulk geometry of the deposit, and productivity requirements to achieve management objectives.

The overall mine plan was developed to meet the capacity of the brownfields milling facilities at Kemess South.

The mine plan is constructed on a subset of the disclosed mineral resource based on metal prices of US$2,400/oz gold and US$4.00/lb copper. The basis for the mine plan inventory is metal prices of US$2,000/oz gold and US$3.75/lb copper, in addition to other factors. A net smelter return (NSR) value is used to determine the cut-off for mill feed, including CA$15.97/t (open pit), CA$56.10/t (underground stoping), and CA$14.10/t (underground development).

The mine plan includes both Inferred and Indicated Mineral Resources, which are summarized in the table below. Inferred Mineral Resources comprise 41% of the Mine Plan.

Table 1-2: Mineral Resources in the Mine Plan

Source of mineralized material	Mineral Resource classification	Tonnes (Mt)	Grade			Contained metal		
			Au g/t	Ag g/t	Cu %	Au koz	Ag koz	Cu Mlb
Main Zone open pit	Indicated	130	0.32	1.14	0.16	1,333	4,769	454
	Inferred	94	0.30	1.02	0.14	905	3,079	296
Underground	Indicated	22	0.93	2.58	0.39	644	1,785	186
	Inferred	9	0.98	2.36	0.40	293	696	80
Kemess Mine Plan		**254**	**0.39**	**1.26**	**0.18**	**3,176**	**10,329**	**1,017**

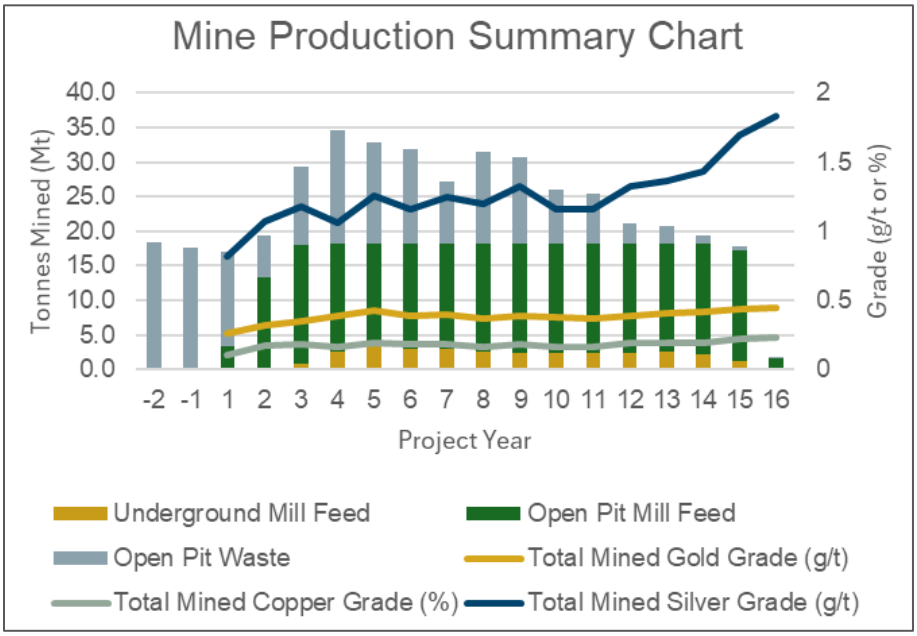
Open pit mining is scheduled to commence 3 years before underground production. The 16-year mine life includes 12 years of concurrent open pit and underground mining. The mining sequence facilitates both a phased pit expansion approach and the inclusion of high-grade underground material early in the mining schedule. Stockpiling of mineralized material is expected to be minimal for Kemess, and any stockpiling needed to meet operational requirements will be stored on the potentially acid-generating (PAG) facility.

Open pit mining peaks at a material movement rate of 32 Mtpa, and averages 23 Mtpa. The planned open pit consists of one pioneering phase for construction material and site access, followed by three production phases. The open pit mine design includes 168 Mt of potentially acid generating waste rock which will require storage in a dry stack facility. The overall tonnage stripping ratio (waste:ore) of the open pit is 0.75. Material is planned to be drilled with two 251 mm diameter rotary drills. Loading will be completed with a fleet consisting of a 30 m^3 electric rope shovel, a 29 m^3 hydraulic face shovel, and a 21 m^3 wheel loader. The haulage fleet includes 140-t class rigid-frame haul trucks. A typical fleet of support and ancillary equipment is included for project development and mine pioneering activities.

Underground productivity averages 2.5 Mtpa or 7–10 stopes per month. The mine design includes six sublevels and average stope dimensions of 20 mW x 20 mL x 35 mH. Cemented paste mine backfill is included in the underground mine plan. The material handling strategy includes ore passes, an underground crusher, and a 3.2 km conveyor to deliver ore to a transfer point where it will be comingled with the open pit material. Over the operating life of mine (LOM), the open pit provides 86% of the planned mill feed for the Project.

Figure 1-3 depicts the combined open pit and underground mining schedule, with annual average grade estimates.

Figure 1-3: LOM production summary



1.7 METALLURGY AND MINERAL PROCESSING

1.7.1 Metallurgical Testwork Samples

Since 2011, there have been 10 metallurgical testwork programs, the most recent being in 2025 (Table 13-1).

Five major mineral types and two waste domains were previously defined in the KUG. All samples submitted for the comminution and flotation test work were selected based on the domain definition shown in Table 1-3.

Table 1-3: KUG rock domains

Domain	Rock type
CP-1	HG monzonite
CP-2	MG monzonite
CP-3	LG monzonite
CP-4	Mixed lithology
CP-5	Upper volcanics
CP-6	Dykes (waste)
CP-7	Hazelton (waste)
Others	Development ore

Test work has been completed on material from both the KUG deposit and the Main Zone. The results establish the comminution design parameters, flotation performance expectations, and metallurgical recovery assumptions applied in the study. The 2025 test program provided additional mineralogy, comminution, and flotation data specifically for the Main Zone material, intended to be mined by open pit and which forms the majority of the planned mill feed.

1.7.2 Leach Test Work

The proposed feed to the leach circuit is the cleaner-scavenger tails. ALS performed cyanide leach bottle roll tests on six different samples of copper first cleaner tails. The leach test work feed samples were produced from locked-cycle flotation testwork, which included a rougher concentrate regrind stage, producing a leach feed grind P80 size between 19 µm and 22 µm. Leach test conditions were with 500 ppm NaCN at pH 11.0, with oxygen sparging to the head space. All tests were performed on 500 g sample at 40% solids.

Sample feed measured gold grades ranged from 0.49 g/t to 1.31 g/t. The leach test results indicate gold recoveries of 55% to 76%.

1.7.3 Recovery Estimates

Recovery estimates for this project are developed from locked cycle test work completed on Main Zone composites in the 2025 program and locked cycle test results for KUG completed in the 2019 program. Main Zone recoveries were derived from master composites representing Broken Zone and Not-Broken Zone material. For KUG, recovery estimates were developed using available 2019 locked cycle test work results for blended composites and individual geometallurgical domains.

Main Zone

Copper recoveries for Main Zone composites range from approximately 82.9% to 89.9%, while gold recoveries range from approximately 53.5% to 58.8%. The testwork produced a copper concentrate grading approximately 23–25% Cu, which is considered reasonable for PEA-level assumptions.

Kemess Underground

In 2019, AuRico determined five main geometallurgical domains from the (KUG feasibility study. For the metallurgical testing programs, blended and individual composite samples were tested in two separate campaigns. Based on the data, the following regression equations were proposed for copper and gold recoveries. These equations can be applied to the underground mineralized material.

- Copper recovery = 101.26 $x^{0.0733}$, where x is copper head grade
- Gold recovery = 7.3536 x + 63.261, where x is gold head grade.

1.7.4 Mineral Processing

The Kemess plant operated from 1999 to 2011 and has been on a care and maintenance program since. There is significant infrastructure already in place, and the existing equipment will be refurbished where possible and supplemented with new and replacement equipment. The plant will process material at a nominal rate of 18.250 Mtpa for 16 years with an average head grade of 0.15% Cu and 0.37 g/t Au. The plant is designed to operate two shifts per day, 365 days per year with an overall plant availability of 92%.

The process plant feed will be supplied by the Main Zone and the Underground deposit and will produce gold rich copper concentrate to be sold to smelters.

As illustrated by the process flow diagram shown in Figure 17-1, mineralized material from the open pit Main Zone will be crushed to a top size of 150 mm and conveyed to a transfer tower at the underground portal. Run-of-mine (ROM) material from the underground mine will also be crushed to a top size of 150mm and conveyed to the underground portal where it will be comingled at the transfer tower with Main Zone material and conveyed to a coarse material stockpile at the plant.

Overland conveying will be utilized to convey mineralized material from the mine site crushers to the process plant. The conveyor routing will be optimized to account for topography and will travel through two tunnels. The downhill segments of the conveyor system will be equipped with reversing drives capable of generating electricity to minimize electrical power usage. Five legs of overland conveyors with variable frequency drives will be constructed with new equipment.

The grinding circuit will consist of two parallel single-train semi-autogenous grind (SAG) and ball mills (Line A and Line B). The target material size introduced into the SAG and ball mill circuits will be 80% passing (P80) 60 mm, or finer, product. Oversize from the SAG discharge screens is recycled back to the SAG mill for reprocessing. The design for the ball mill circuit discharge and feed to flotation is a product size of P80 <42 µm (0.04 mm).

The refurbished rougher flotation circuit will produce a rougher concentrate that will be directed to the first stage regrind circuit. The rougher concentrate will be classified in a regrind cyclone cluster, with coarse material sent to the refurbished regrind ball mill which will target a regrind product of approximately 40 µm and a portion treated by a refurbished gravity concentrator. The refurbished regrind cyclone overflow will feed the first cleaner flotation (refurbished cells) followed by cleaner scavenger flotation (refurbished cells).

Concentrate from the first cleaner and cleaner-scavenger stages will be further reduced in size in a new high-intensity grinding mill (HIGmill™) equipped with a Cyclopac, targeting a product P80 of <20 µm. The HIGmill product will be transferred to the secondary cleaner flotation, which will consist of a new Jameson Cell. Secondary cleaner concentrate will advance to the third cleaner flotation column (refurbished) and then to the refurbished scavenger flotation column for final upgrading.

Tailings from the cleaner-scavengers will be sent to a conventional CIP leaching plant. The ADR portion of the plant will produce gold as doré and the tailings from the leach plant will discharge into the old Kemess south pit for the LOM.

The resulting final flotation concentrate is expected to target approximately 23% Cu, with these targets adjustable depending on metal prices and smelter terms. Gold recovered in the gravity concentrator will be directly fed to the final concentrate thickener, bypassing the cleaner circuit.

Final flotation concentrate will be thickened, dewatered, and pressure-filtered to a target moisture content of 8%, stockpiled, and then trucked to the rail loadout facility in Mackenzie, BC. The concentrate will then be railed to North Vancouver, where it will be loaded onto ships and transported to purchasers.

Rougher tailings and the concentrate clarifier overflow will be combined into a single tailings stream and pumped to the TSF. No separate leach tailings will be generated, as the new CIL–ADR circuit processes cleaner scavenger concentrate. Tailings deposition will be split between the two facilities, the old Kemess south pit and TSF, and will vary over the life of mine.

1.7.5 Leaching of Gold

Testing has identified an opportunity to enhance gold recovery by leaching the cleaner-scavenger tails, which contain between 0.49 g/t and 1.15 g/t Au. The feed is pyrite-rich, and study work has shown that a conventional cyanide leach with an ADR is the most effective way to improve global gold recovery.

Ground slurry from the cleaner scavenger tails reports to a thickener which underflow slurry is pumped to the oxygen-rich leaching circuit where NaCN is introduced at high pH. The leached slurry enters the CIP circuit where activated carbon becomes loaded with gold. Loaded carbon is periodically withdrawn from the first CIP tank and transferred to the loaded carbon wash screen. Loaded carbon batches of approximately four tonnes are treated in the acid wash vessel to remove inorganic deposits. Spent acid

and rinse water are discharged to the tailings thickener feed box. After washing, carbon is transferred to the elution column using process water.

Gold stripping is performed using a high-temperature pressure process. A barren solution containing NaOH and NaCN is circulated through the elution column at elevated conditions of 135°C and >240 kPa. The solution is heated via heat exchangers and water heaters, and pressure is controlled to prevent boiling. The pregnant eluate exiting the column is directed to electrowinning cells.

Pregnant solution from elution is processed in electrowinning cells equipped with cathodes and anodes. Gold-bearing sludge is removed from cathodes and dewatered in a filter press. The sludge is dried in an oven, mixed with fluxes, and smelted in an induction furnace to produce doré bullion.

1.7.6 Reagents and Process Systems

The reagent handling system will include storage, mixing, and metering equipment as needed for each of the required reagents. Each reagent will be located in a dedicated reagent containment are to prevent environmental contamination and mixing of incompatible reagents. Reagent and consumable consumption estimates were developed using anticipated material characteristics, industry benchmarking, and initial test work results.

Separate systems are installed to deliver process, fire, gland seal and potable water. The compressed air system at the plant will be refurbished. The total installed electrical load for the process plant is estimated at 66 MW, with an annual power consumption estimated of 454 GWh/a.

1.8 PROJECT INFRASTRUCTURE

1.8.1 Off-site Roads, Airstrips and Loadouts

The 380 km access road from MacKenzie to Kemess is being maintained and used on a seasonal basis. The road will be used for truck transport of concentrates to the rail head at Mackenzie and delivery of materials and supplies to the mine.

The 16.5 km forest service Cheni Mine Road is an all-season gravel road connecting the Kemess North site to the Kemess Main Road turn-off. To accommodate non-loaded mining trucks, two bridges along the Cheni Mine Road require upgrading in addition to minor clearing and widening activities.

A 1.6 km all-weather air strip is maintained for passenger aircraft and transport planes. A twin-engine Beech aircraft was routinely used to ferry personnel on and off site during the operation of the Kemess South Mine.

In Mackenzie, Centerra retains a trans-shipment facility next to a rail spur, for shipment of Mount Milligan concentrates, which will also accommodate Kemess shipments. Grinding media consumables are

received by rail in MacKenzie for transhipment to the mines. The loadout facility at Mackenzie will need to be upgraded to handle increased volumes of concentrate.

1.8.2 Site Infrastructure

The Kemess South Mine site previously supported a 50,000 tpd open pit mining and milling operation including an administration building, assay and metallurgical labs, a truck shop, security facilities, and a personnel camp. Capital costs have been included for the renovation and expansion of camp services as well as the development of a new camp to support operations for the LOM.

A new infrastructure pad is required at the Kemess North site to accommodate a truck shop for maintenance servicing and minor repairs, a fuel station, an emergency station, and offices. Other mine site infrastructure will comprise mine administration and maintenance facilities, offices, warehouse, a communications network, an energy management system, service mobile equipment, security facilities, and information technology (IT) installations.

1.8.3 Tailings Storage

During previous Kemess Mine operations approximately 213.4 Mt of tailings were stored at the Kemess South TSF, with an additional 17.4 Mt of tailings directed to the mined-out portion of the Kemess South open pit (renamed the Kemess Underground, or KUG, TSF). The Kemess South TSF was constructed as a modified centreline embankment over 12 stages between 1996 and 2010.

A 2025 study focused on identifying and designing suitable tailings storage solutions utilizing the existing Kemess South TSF and the KUG TSF for the deposition of an additional 261 Mt of tailings and 158.5 Mt of potentially acid-generating (PAG) waste rock.

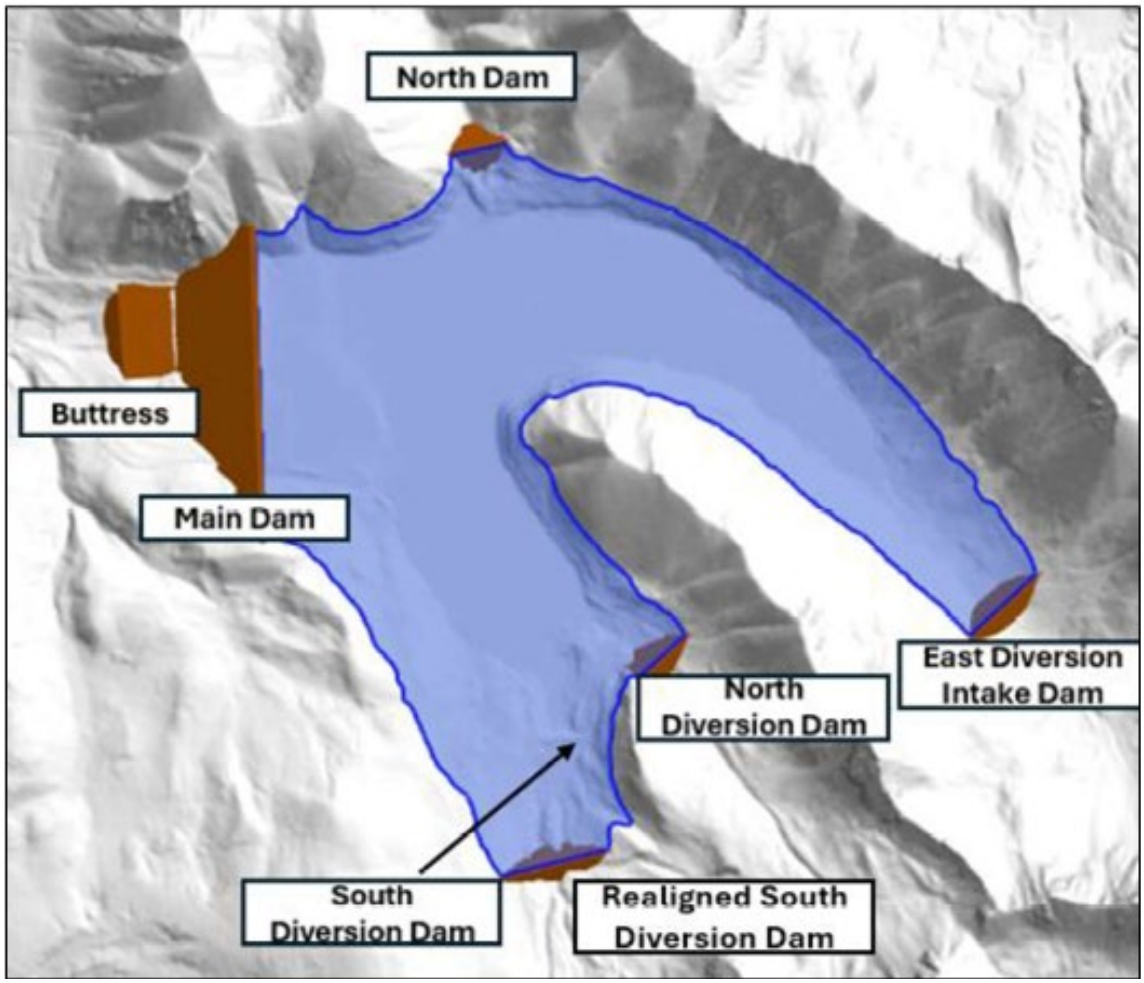
Tailings Characterization

Two types of tailings have been designated for the project, rougher tailings and cleaner tailings. Rougher tailings are expected to comprise approximately 85% of the total tailings volume and are anticipated to be non-acid generating (NAG). The remaining 15% are classified as cleaner tailings and are expected to be PAG.

Kemess South TSF

The existing Kemess South TSF main embankment will be raised 26 m to a final elevation of 1,537 masl to store approximately 122 Mm^3 of operating pond, and 3 m freeboard. The main embankment will be raised using the centreline construction method, with a low-permeability glacial till keyed into the existing dam. The Kemess South TSF main embankment downstream shell and stabilizing buttress will be comprised of cyclone sand produced from rougher tailings. Figure 1-4 shows a model rendition of the Kemess South TSF containment dams.

Figure 1-4: Kemess South TSF model showing containment dams



A new North Saddle Dam will be required along the north side of the facility to provide adequate containment. This structure will be constructed as a downstream raise rockfill shell dam with a low-permeability core.

The increase in the facility size has required an upgrade to the existing Kemess South TSF diversion system. This upgrade includes raising the existing South and North Diversion Dams using downstream and centreline methods, respectively, as well as adding a centreline raise to the East Diversion Dam. All dams will consist of rockfill shells with low-permeability cores and appropriate transition materials on both sides.

In addition, new South and North Diversion Channels and an East Diversion Conduit will be constructed, all graded to convey flows from the East and North Diversions toward the South Diversion, where they will be discharged to Kemess Creek.

The surficial geology in the vicinity of the Kemess South TSF is comprised of bedrock, varved glaciolacustrine sediments, silty sands and gravels, and glaciofluvial deposits. Results of stability analyses indicate the TSF embankments achieve the required factor of safety.

Closure of the Kemess South TSF will include reinstating a spillway channel on the left abutment of the Main Dam, tying into the existing, unaffected spillway. The spillway invert will be set to maintain a water cover over the tailings to an elevation of 1,533 masl. Preliminary closure plans for the Kemess South TSF includes placement of a vegetated cover over the surface and downstream embankment slopes to promote long-term stability and rehabilitation to blend in with the surrounding area.

Kemess Underground TSF

The KUG TSF will utilize the existing Kemess South open pit to store approximately 63 Mm^3 of tailings, a 4 Mm^3 operating pond, and a three-meter freeboard below the existing pit rim elevation of 1,260 masl. The KUG TSF design includes an East Dam and a South Saddle Dam, each constructed to an elevation of 1,295 masl.

The KUG TSF is developed from the existing Kemess South open pit and will require minimal modifications to its current configuration. Tailings lines will be routed from the mill to the TSF and deposited within the pit via subaqueous deposition. Tailings and water management will be conducted through a barge system that transfers water from the pit to the plant site. The rate of rise in the pit is expected to be approximately 10 m in elevation per year for the first three years, followed by an average of 2 m per year until the end of deposition.

The closure plan for the KUG TSF is to restore the site to as natural a state as practicable, protect the downstream environment, and effectively manage surface water. The preliminary closure concept includes the construction of an emergency spillway at the eastern low point of the facility. The spillway will maintain a consistent water level within the TSF while allowing surplus water to be safely released to the environment. The KUG TSF impoundment is anticipated to require a water cover to mitigate oxidation of cleaner tailings.

1.8.4 Tailings Pipeline

At mill startup until Year 3, the plant tailings will flow by gravity in a pipeline from the mill to an existing mixing box in pump house 1. A new tailings pipeline will transport whole tailings to the KUG TSF.

From operating Year 4, when the leach plant becomes operational, high pyrite tailings from the leach circuit will be pumped directly into the KUG TSF for the remaining mine life.

Rougher flotation tailings will flow from the process plant to pump house 1 to be pumped to the Kemess South TSF. To reach the TSF, two existing 26 inch tailings pipelines (one operational and one on

standby) and all the necessary pumps will be replaced or refurbished, depending on equipment condition.

1.8.5 Water Management

Upon operations restart, reclaim water for mill operations will be sourced from the KUG TSF where the existing reclaim barge and pumps will be refurbished to restore operational reliability. Reclaim water will flow through the existing twin pipelines to the existing reclaim water transfer station. Piping infrastructure will be refurbished as required.

The contact water management system developed for the existing South Waste Rock Storage Facility (WRSF) will remain in operation throughout the life of mine. Runoff and seepage generated from the waste rock stockpile will be collected and conveyed to the Water Treatment Plant for treatment prior to discharge. For the North WRSF, contact and seepage water will drain to dedicated collection ponds and be pumped to the same Water Treatment Plant.

1.9 ENVIRONMENT, COMMUNITY AND PERMITTING

New environmental studies are required to support the environmental assessment for permitting the KRP. These studies include, but are not limited to, water quality and quantity, fisheries and aquatic resources, archaeology, soils, vegetation, and wildlife. Field work to complete the environmental studies commenced in 2024. with the aim to have baseline collection completed by the end of 2027/early 2028.

Modeling of surface water and water quality has confirmed that all contact water from the Main Zone Area – including the Main Zone Pit and the PAG WRSF – will be captured and routed south to the existing treatment infrastructure, ensuring no impact on Amazay Lake. The water balance model predicts that annual contact water volumes will rise from 1.1 Mm^3 in Year 1 to a peak of approximately 2.5 Mm^3 by Year 13.

To address predicted metal concentrations from mining activities, water will be pumped to the south area Water Treatment Plant, further study work is planned assess any impacts in the East Cirque Creek catchment. Also, a Site Performance Objective is currently being developed to establish long-term selenium permit limits for Waste Rock Creek.

The Kemess site possesses an extensive biological database for fisheries and aquatic resources, with continuous data collection dating back to 1992 for sediment quality and fisheries resources. This long-term monitoring satisfies commitments under the Fisheries Compensation Agreement and the Fish and Aquatic Effects Monitoring Plan.

The permitting strategy for Kemess centres on amending the project plan of operation from an underground block cave mine to an open pit and underground blasthole operation. This approach allows

the project to utilize “shovel ready” permits for immediate field activities – such as refurbishing the mill, constructing the access tunnel, and dewatering the KUG TSF – while the more complex major amendments are processed.

Centerra will employ a Concurrent Permitting model, successfully utilized for the Mount Milligan 2035 LOM expansion, which emphasizes deep interagency coordination to reduce regulatory uncertainty. The strategy contemplates that the Impact Assessment Agency of Canada (IAAC) will be integrated into the concurrent model alongside provincial regulators, such as the Ministry of Mines and Critical Minerals (MCM) and the Ministry of Environment and Parks (ENV). This synchronized approach ensures that federal oversight under the *Impact Assessment Act* is managed in parallel with provincial *Mines Act* and *Environmental Management Act* (EMA) amendments.

The site currently holds Mines Act Permit M-206, EMA Effluent Permit PE-15335, and Air Emission Permit PA-109392. The existing regulatory framework for the Kemess site provides a robust foundation of “shovel-ready” authorizations that allow critical field activities to proceed while KRP-specific amendments are processed. The Project is currently authorized to perform the dewatering of the KUG TSF (the former Kemess South pit) at a defined discharge rate.

Both the Kemess South TSF and KUG TSF are permitted for the deposition of tailings.

The proposed PAG stockpile will be a lined, sub-aerial dry-stack facility located in the East Cirque Valley, northeast of the proposed Kemess Main Zone open pit designed to provide secure and permanent storage for approximately 158.5 Mt (72 Mm^3) of PAG waste rock over a 17-year operating life. The WRSF design features a sophisticated dual-layer water management system designed to capture seepage and meteoric water – a collection system above the liner to collect meteoric water percolating through the waste rock and a drain system beneath the liner as a contingency system. Both systems direct seepage through a network of perforated pipes toward the North Collection Pond from which water will be pumped to a treatment facility.

An Early Works Permit application will seek authorization for North Access Road upgrades, bunkhouse renovations, and the realignment of one of the three approved decline tunnels. A Major Works Permit application will seek a combined Environmental Assessment Certificate (EAC)/Federal Decisions Statement and Mines Act/EMA amendment to cover the Main Zone and Nugget Pits, the PAG WRSF, the new leach plant, and the expanded Kemess South TSF.

Each of the Works permit applications are expected to take 12 months for approval.

At mine closure, while the ultimate goal is passive treatment, the KRP plans, and has allocated costs, for active treatment during the transition phase, including KUG TSF water treatment for five years post-

operations and the treatment of contact water from the WRSF and pit highwalls at the Main Zone mining area until reclamation is complete and water quality meets permit criteria for release.

Under the existing Impact Benefit Agreement (IBA) with the Tse Keh Nay (Takla, Kwadacha, Tsay Keh Dene Nations), the Environmental Management Committee (EMC) serves as the primary mechanism for collaboration on all KUG Project-related environmental matters. Takla, Kwadacha, Tsay Dene Nations members of the EMC participate in the collaborative development of permit applications and management plans, ensuring that Traditional Knowledge and Takla, Kwadacha, Tsay Dene Nations values are integrated into the project design before submission to government agencies.

Centerra maintains a proactive and transparent relationship with its Indigenous partners Takla, Kwadacha and Tsay Keh Dene First Nations, and Gitxsan Nii Gyap First Nations. A comprehensive IBA was executed in May 2017, establishing a long-term framework for financial benefits, employment, and environmental oversight with Takla, Kwadacha and Tsay Keh Dene.

Centerra also maintains an active agreement with the Gitxsan Nii-Gyap First Nation that is specifically designed to provide benefits for permit review, cultural protection, and general capacity building. The relationship of the Project with local land users is formalized through the Trapline Holders Settlement and Release, an agreement originally executed on December 18, 2014, with the registered holders of Trapline No. 0739T006.

A number of committees have been struck to coordinate collaborative governance, community outreach and communications with regional residents.

1.10 MARKETS AND CONTRACTS

Upon achieving commercial production, the Kemess Mine will produce gold and silver doré, and a copper and gold concentrate.

Copper concentrate produced through a conventional flotation process will be sold and shipped to smelters. Copper grade of the concentrate is projected to be around 21%, which is expected to result in a payable factor for copper of about 94.8%, making the concentrate from Kemess Mine readily marketable.

The doré bars will be shipped to third-party refiners for final refining and sale. Based on common industry practices, payable metal content from doré sales is estimated to be approximately 99.8% for gold and 90% for silver, relative to the contained metal in the doré.

1.11 COST ESTIMATES

Cost estimates for the Kemess Project were developed with reference to similar open-pit and underground mining operations. Any estimates made in Canadian dollars were converted to constant US dollars applying an exchange rate of 1.38 (CA$:US$) as of December 31, 2025.

1.11.1 Capital Costs

Capital and operating cost estimates are preliminary in nature and prepared to AACE Class 5 level of accuracy (-50%/+100%) appropriate for a PEA study. The estimates are based on conceptual mine plans, benchmarked data, and assumptions considered reasonable for this stage of study.

A summary of total LOM capital and operating costs are detailed in Table 1-4.

Table 1-4: LOM capital and operating costs

Cost summary ($ M)	LOM total[1]
Operating costs	
Open pit mining	1,063
Underground mining	966
Processing	1,558
General and administration	975
Transportation	422
On-site operating costs	**4,985**
Treatment and refining	131
Selling and marketing	98
Subtotal – operating costs	**5,213**
Capital costs	
Initial non-sustaining capital costs [NG]	771
Expansionary non-sustaining capital costs [NG]	277
Sustaining capital costs [NG,2]	595
Subtotal – capital costs	**1,643**
Total costs	**6,858**

Notes: (1) Totals may not sum precisely due to rounding. (NG): initial non-sustaining capital costs, expansionary non-sustaining capital costs, sustaining capital costs are non-GAAP financial measures. (2) Does not include capital lease payments of $229 million which are included in the economics and operating costs above.

LOM capital costs, summarized in Table 1-5, are estimated to total $1.6 billion, comprising the following three distinct phases:

- Initial non-sustaining capital costs – $771 million
- Expansionary non-sustaining capital costs – $277 million
- Sustaining capital costs – $595 million.

Initial non-sustaining capital costs, expansionary non-sustaining capital costs, and sustaining capital costs incorporate 40% for indirect costs and 30% contingency applied to construction items.

Table 1-5: LOM capital costs[1] summary

Capital costs ($ M)	Initial non-sustaining	Expansionary non-sustaining	Sustaining capital costs	Total capital costs
Process plant	96	64	37	197
Open pit conveyor system	65	-	-	65
Open pit crusher	25	-	-	25
Open pit infrastructure and other	29	-	25	54
PAG storage facility	24	-	11	34
General site infrastructure	27	-	11	38
Tailings facility	9	-	89	98
Underground infrastructure	-	43	56	99
Underground development	-	59	38	97
Underground conveyor system	-	12	7	19
Paste backfill plant	-	-	75	75
Mobile fleet overhauls/replacements	-	3	101	103
Subtotal Infrastructure directs	**275**	**180**	**448**	**902**
Construction indirects	110	47	72	229
Contingency	115	50	76	241
Subtotal Infrastructure costs	**500**	**277**	**595**	**1,372**
Capitalized open pit pre-stripping	131	-	-	131
Capitalized pre-production G&A	101	-	-	101
Open pit conveyor tunnel	39	-	-	39
Total indirect and other costs	**271**	**-**	**-**	**271**
Total Capital Costs	**771**	**277**	**595**	**1,643**

Notes: (1) Initial non-sustaining capital costs, expansionary non-sustaining capital costs, sustaining capital costs are non-GAAP financial measures. (2) Totals may not sum precisely due to rounding.

1.11.2 Mining and Processing Schedules

The LOM mining schedule is presented in the Table 1-6, followed by the LOM processing, concentrate and metals production schedule in Table 1-7.

Table 1-6: LOM mining production schedule

Mine Production	Units[1]	LOM Total	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046
Open Pit mine production																					
OP ore mined	Mt	**224**	-	-	-	3	14	17	16	15	15	15	16	16	16	16	16	16	16	16	2
OP waste mined	Mt	**168**	-	18	18	14	6	11	16	15	14	9	13	13	8	7	3	3	1	1	0
Open Pit material mined	**Mt**	**392**	**-**	**18**	**18**	**17**	**20**	**29**	**32**	**30**	**29**	**24**	**29**	**28**	**23**	**23**	**19**	**18**	**17**	**17**	**2**
UG mine production																					
UG ore mined	Mt	**31**	-	-	-	-	0	1	3	3	3	3	3	2	2	2	2	3	2	1	-
UG waste mined	Mt	**2**	-	-	-	1	1	0	0	0	0	0	-	-	-	-	-	-	-	-	-
UG material mined	**Mt**	**33**	**-**	**-**	**-**	**1**	**1**	**1**	**3**	**3**	**3**	**3**	**3**	**2**	**2**	**2**	**2**	**3**	**2**	**1**	**-**
Total mine production																					
Total ore mined	Mt	**255**	-	-	-	3	14	18	18	18	18	18	18	18	18	18	18	18	18	17	2
Total waste mined	Mt	**170**	-	18	18	14	7	12	16	15	14	9	13	13	8	7	3	3	1	1	0
Total material mined	**Mt**	**425**	**-**	**18**	**18**	**18**	**21**	**30**	**35**	**33**	**32**	**27**	**31**	**31**	**26**	**25**	**21**	**21**	**19**	**18**	**2**
Rehandle material moved	Mt	**2**	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total material moved	**Mt**	**427**	**-**	**18**	**18**	**18**	**23**	**30**	**35**	**33**	**32**	**27**	**31**	**31**	**26**	**25**	**21**	**21**	**19**	**18**	**2**

Notes:
1. "Mt" refers to millions of tonnes.
2. Totals may not sum due to rounding.

Table 1-7: Processing, concentrate and metals production schedule

Processing Production	Units[1]	LOM Total	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046
Ore processed	**Mt**	**255**	**1**	**16**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**17**	**2**
Gold feed grade	g/t	**0.39**	0.26	0.32	0.35	0.39	0.42	0.39	0.40	0.37	0.39	0.38	0.37	0.39	0.41	0.41	0.44	0.44
Silver feed grade	g/t	**1.26**	0.82	1.07	1.18	1.06	1.25	1.16	1.25	1.20	1.32	1.16	1.16	1.33	1.36	1.43	1.70	1.83
Copper feed grade	%	**0.18%**	0.10%	0.17%	0.18%	0.16%	0.19%	0.18%	0.18%	0.17%	0.18%	0.16%	0.17%	0.19%	0.19%	0.20%	0.22%	0.24%
Contained gold processed	Koz	**3,176**	11	165	204	228	248	226	234	215	229	220	217	230	239	243	242	24
Contained silver processed	Koz	**10,336**	36	544	685	623	736	682	733	704	777	681	679	778	800	840	941	98
Contained copper processed	Mlb	**1,017**	3	59	73	65	76	73	73	66	73	66	67	76	76	79	83	9
Gold recovery	%	**74.5%**	53.3%	53.3%	69.6%	73.8%	75.9%	75.9%	77.0%	76.7%	78.0%	77.5%	77.6%	77.0%	77.6%	76.6%	75.3%	74.3%
Silver recovery	%	**50.1%**	45.0%	45.0%	46.3%	51.2%	52.2%	52.5%	52.0%	51.2%	50.5%	51.6%	51.0%	50.5%	50.4%	49.7%	47.1%	45.0%
Copper recovery	%	**88.2%**	81.4%	81.4%	83.3%	86.2%	87.1%	87.3%	88.8%	89.5%	90.3%	90.0%	90.4%	90.1%	90.2%	90.0%	89.5%	89.3%
Gold recovered	Koz	**2,376**	6	88	142	168	189	172	180	165	178	171	168	177	185	186	182	18
Silver recovered	Koz	**5,171**	16	245	317	319	384	358	381	360	393	351	347	393	403	417	443	44
Copper recovered	Mlb	**898**	3	48	61	56	66	64	65	59	66	59	61	68	69	71	75	8
Dry concentrate produced	Kdmt	**1,939**	6	105	131	121	143	138	140	128	142	128	131	148	149	153	161	17
Gold payable produced	**Koz**	**2,323**	**6**	**86**	**139**	**166**	**185**	**168**	**176**	**162**	**175**	**167**	**165**	**174**	**181**	**182**	**179**	**14**
Silver payable produced	**Koz**	**4,654**	**15**	**220**	**285**	**287**	**346**	**322**	**343**	**324**	**353**	**316**	**312**	**353**	**363**	**376**	**399**	**40**
Copper payable produced	**Mlb**	**851**	**2**	**46**	**57**	**53**	**63**	**60**	**61**	**56**	**62**	**56**	**57**	**65**	**65**	**67**	**71**	**7**

Notes:
1. "Mt" refers to millions of tonnes; "koz" to thousands of ounces; "Mlb" to millions of pounds; and "Kdmt" to thousands of dry metric tonnes.
2. Totals may not sum due to rounding.

1.11.3 Operating Costs

Operating costs are estimated to total $5.2 billion over the LOM and include open pit mining, underground mining, processing, general services and administration (G&A), transportation, treatment and refining, and selling and marketing costs. A summary of operating and all-in sustaining costs (AISC) is presented in Table 1-8:

Table 1-8: LOM operating costs

Cost summary	Average LOM ($/t processed)
Operating costs	
Open pit mining	4.17
Underground mining	3.79
Processing	6.11
G&A	3.83
Transportation	1.66
On-site operating costs	19.56
Treatment and refining	0.51
Selling and marketing	0.39
Total operating costs	**20.46**

The LOM operating cost per tonne processed is estimated to average $20.46. Gold production costs are estimated to average to $1,401 per gold ounce sold over the LOM. The AISC estimate includes sustaining capital costs, sustaining lease payments, ARO accretion expense and by-product credits for net copper and silver sales, and is estimated to average $971 per gold ounce sold over the LOM. AISC is projected to be lower than the average gold production cost per ounce, as the total of copper and silver by-product revenue credits are anticipated to exceed sustaining capital expenditures.

The Kemess PEA outlines a development approach in which open pit mining begins first, followed by the start of underground production approximately two years later. Once underground production commences, both mining methods operate concurrently for the remainder of the projected 15-year mine life.

Mining operations will begin with waste pre-stripping at the Kemess Main open pit with production from the open pit planned to commence two years later in addition to underground development. Underground production will start approximately two years later. Production from underground is expected to achieve 0.8 Mt in Year 2 and 2.5 Mt in Year 3. Prior to the start and ramp-up of underground operations, the open pit will supply process plant feed at a nominal average rate of 40,000 tpd, or 15 Mtpa from Years 2 to 3. Once both open pit and underground mines are operating concurrently from late in Year 2, combined mining activities will provide process plan feed at an average rate of approximately 50,000 tpd, or 18 Mtpa from Year 3 onward.

1.11.4 Mining

A summary of open pit mining costs is presented in Table 1-9:

Table 1-9: LOM open pit mining costs

Cost summary	Average LOM ($ / open pit tonne mined)
Open pit mining costs	
Direct open pit mining	2.81
Open pit conveying and materials handling	0.22
Total open pit mining costs	**3.03**

The LOM open pit mining unit cost estimate of $4.17/t processed equates to approximately $3.03 per open pit tonne mined which includes the direct unit cost of $2.81/t mined and $0.22/t mined (transported via open pit conveyor and rehandling). The mining rate of $2.81/t mined was developed primarily with the reference to comparable Canadian open pit mining operations, including the Company's Mount Milligan Mine, adjusted for incremental costs and taking account of the location of the Kemess mine and related logistics.

A summary of underground mining costs is presented in Table 1-10.

Table 1-10: LOM underground mining costs

Cost summary	Average LOM ($ / underground tonne mined)
Underground mining costs	
Direct underground mining	23.51
Underground operational development	7.59
Underground mineralized material transportation	0.38
Total underground mining costs	**31.48**

The LOM underground total mining cost is estimated at $31.48 per underground tonne processed. The total underground mining cost includes direct mining costs, operational development, backfill placement, and conveying. On a per tonne processed basis, total underground mining unit cost equates to approximately $3.79/t.

1.11.5 Processing

After the processing plant refurbishments and upgrades, the Kemess plant is expected to achieve daily throughput production of 50,000 tpd, or 18 Mtpa. The updated flowsheet includes the construction of the leach plant, which is expected to increase overall gold recovery by approximately 14%, enhancing the project's economics. In addition to improving recovery, it is expected to provide flexibility by enabling the processing of mineralized material from potential satellite deposits in the future.

Processing cost estimates are based on the existing Kemess process plant and the refurbishments required to return it to operational status, supported by benchmarked data, and assumptions considered reasonable for this stage of assessment. The confidence level of the estimates of processing costs is enhanced by the fact that the Project already has the major processing facilities, structural components, and plant layout in place. Processing costs were developed from basic cost elements including labour, reagents, consumables, power rates, water treatment costs, and equipment productivities.

The LOM processing cost is estimated to average approximately $6.11/t processed. The processing cost includes tailings pumping costs and leach plant operating costs. Electrical power costs were estimated based on expected consumption and power rates calculated by a third-party consultant from first principles using an estimated project load list. Consumption estimates and pricing for grinding media and reagents are based on metallurgical test work results, and internal procurement data collected from the Mount Milligan Mine, supplemented with historical data from similar operations in British Columbia, and adjusted for transportation costs to the Project site.

1.11.6 General and Administration Costs

G&A costs include all site-based support services required to operate the Kemess Mine and excludes any allocation of corporate G&A. These costs cover camp operations, flight logistics, site security, water treatment, environmental services, insurance, property taxes, permitting expenses, IBA commitments and other general and administrative functions not captured within mining, processing, treatment, or transportation cost categories.

The average G&A cost is estimated at approximately $3.83/t milled over the LOM. The G&A costs include site general costs of $2.82/t milled and the IBA commitments cost that contributes approximately $1.01/t milled. Water treatment costs included in the G&A costs are estimated to cost $3.6 million per year starting from 2037 to 2045, contributing approximately $0.13/t milled over the LOM.

Labour costs were developed based on projected workforce requirements and expected labour rates for northern British Columbia.

Insurance, permitting, and property taxes were benchmarked to the current costs at the Company’s Mount Milligan Mine, reflecting comparable jurisdictional and operational conditions. Third-party service costs including camp management, catering, charter flight operations, site security, water treatment, environmental monitoring, and other professional services were estimated using vendor quotes obtained for major cost categories and based on the projected scope of support services required during operations.

1.11.7 Closure and Post-Closure Costs

Closure and post-closure activities will begin progressively following the end of production, with the majority of physical reclamation and earthworks planned for the two years immediately following cessation of production operations.

On an undiscounted and uninflated basis, LOM closure and reclamation cash costs are estimated to be approximately $180 million. For purposes of the economic analysis, reclamation expenditures were discounted back to January 1, 2046, representing both the end of mine operations and the assumed start date for mine closure, and incorporated into the cash-flow model, with the discounted cost to January 1, 2046 representing approximately $100 million. This amount is then further discounted to the Project valuation date of January 1, 2028 within the cash-flow model, consistent with the treatment of all other cash flows.

While ongoing post-closure activities for monitoring, water management, and maintenance of closure works extend beyond 2048, the present value of these expenditures is fully captured within the discounted closure estimate used in the economic analysis.

1.12 ECONOMIC EVALUATION

Economic estimates are preliminary in nature and have been prepared to an AACE Class 5 level of accuracy (-50%/+100%), appropriate for a PEA study.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

The economic analysis of the project was conducted using the following assumptions and basis:

- Project economics are based on a valuation date of January 1, 2028. The economic assessment employs a discounted cash flow (DCF) approach, with cash flows assumed to occur at the mid-year of each period. The net present value (NPV) is calculated by discounting the LOM cash flows from January 1, 2028 through the end of the LOM at a discount rate of 5%. Economics include the time value of money benefit of pushing out $69 million of care and maintenance and closure costs to the end of the LOM.
- Open pit mining activities scheduled from January 1, 2029, followed by underground development starting in 2031. Ore processing will begin in late 2031 and continue to the end of mine life in 2045. The mine life, counted from the start of ore processing in late 2031 to 2045, is estimated to be 15 years.

- All costs presented are in constant United States dollars as of December 31, 2025, with no price inflation or escalation factors applied.
- The metal price assumptions for gold, copper, and silver, and the US dollar to one Canadian dollar exchange rate used in the evaluation of the project economics are as follows:
  - Gold price per troy ounce: $3,000
  - Copper prices per pound: $4.50
  - Silver prices per troy ounce: $37.50
  - US dollar to one Canadian dollar exchange rate: $1.38.
- The silver produced is subject to a stream arrangement with Triple Flag. The impact of the stream arrangement is fully incorporated into the project economics.
- Working capital for the mine is assumed not to change significantly over the LOM and is not modelled in this economic analysis.
- No salvage values are assumed for the capital equipment at the end of mine life.
- Transportation costs include estimated delivery costs of gold and silver doré, and copper concentrate to customers.

1.13 TAXATION

The determination of taxes involves significant estimation and judgment requiring a number of assumptions. The actual taxes payable for the project will be subject to assessments by taxation authorities who may interpret tax legislation differently. The after-tax cash flow is based on best estimates by Company management of the probable outcome of these matters.

1.13.1 Corporation Income Taxes

Based on the pricing assumptions noted above, the Kemess Mine is expected to be subject to Canadian federal and British Columbia provincial corporate income taxes at the combined statutory rate of 27%. Corporate income tax liabilities can be partially reduced by available tax deductions, which are expected to partially offset taxable income. These deductions include:

- Exploration expenditures allowed to be claimed discretionarily at 100%, limited to the mine's taxable income
- Pre-production development expenditures allowed to be claimed discretionarily at 30% of the year-end balance
- Initial and sustaining capital expenditures generally allowed to be claimed discretionarily at 25% of the year-end balance

- Net operating loss carry-forward allowed for up to 20 years
- Provincial mining taxes.

Following the utilization of these deductions, the Kemess Mine is expected to be subject to corporate income taxes at the statutory rate.

1.13.2 British Columbia Mining Taxes – Provincial

The mine will be subject to the greater of two different taxes: either 2% tax on net current proceeds (net revenue less operating costs) or 13% tax on net revenue (net revenue less operating costs and capital expenditures). Based on the pricing assumptions noted above, the mine is expected to pay the 2% net current proceeds tax for approximately the first half of its life, as the Company expects to have sufficient deductions and credits during that period to offset the 13% tax on net revenue, while the mine is expected to pay the 13% tax on net revenue for the second half of its life. In lieu of allowing a deduction of debt financing costs, the net revenue can be reduced by an investment allowance which is earned on expenditures incurred to the extent they have not yet been deducted.

1.14 LIFE-OF-MINE CASH FLOW FORECAST

The net undiscounted cash flows for the Kemess Mine from January 1, 2028 to the end of 2048 are estimated at $2,329 million, as presented in Table 22-2. The after-tax NPV of the LOM cash flow, discounted at 5% is estimated at $1,094 million.

1.15 STUDY CONCLUSIONS

1.15.1 Geology and Mineral Resources

Geological Model

The QP considers the lithological model to be robust for both Kemess Main Zone and Kemess South models, as the principal lithological units are characterized by clear and readily distinguishable geological and physical contrasts in core and logging data. These contrasts support consistent lithological interpretation and domain definition at the current level of study. However, lithological interpretations may be refined as additional drilling is completed.

The Mineral Resource estimate for the Kemess Project is subject to uncertainties typical of a PEA-level study. Geological risks include uncertainty in the continuity of mineralization between drill holes, particularly in areas of wider drill spacing, along interpreted structural controls, and at depth, which may affect the confidence in resource classification and the geometry of the mineralized domains.

Resource Estimation

The Kemess Main Zone open pit and underground resource estimate carries inherent risks typical of early-stage projects, particularly due to the absence of production data. Key assumptions and confidence levels are primarily supported by geological interpretation and variogram-based continuity in the absence of production data to support reconciliation. Uncertainty remains until mining begins, as actual grade distribution, dilution, and recovery may differ from model predictions.

The Kemess South resource estimate benefits from historical operational and production data. Areas of the model that have been depleted show reasonable reconciliation with the current geological interpretation, lending further confidence to the estimate in previously mined domains.

Sampling-related risks, including potential bias, representativity, and density assumptions, are also recognized. The mineralization has been modelled as a continuous, single-phase system and is not expected to present significant additional geological complexity; however, it is recommended that the estimate be further validated through ongoing drilling, model updates, and eventual production performance data to enhance confidence in future reporting.

1.15.2 Mining

The PEA uses a concept of an open pit and an underground mine to optimally extract the maximum mineralized material from the Kemess Main Zone deposit. Advanced engineering studies will seek optimized techniques and sequencing.

The main risks to the mine plan are geotechnical, given the unknowns associated with this level of study. Underground infrastructure locations have evolved from the previous block cave feasibility study. While historic information is available, the review and re-interpretation of this information is ongoing. Gaps in data and interpretation will need to be closed at the next phase of study.

The open pit mine is located in a topographically challenging area with risks that include a talus slope, natural rockfalls and snow avalanches. While this PEA has concluded that removing the loose material of the talus slope is the best way to manage that risk, LOM geohazard management will be required.

Infrastructure congestion is another risk that will require additional study at the prefeasibility study level. Surface infrastructure locations will need to be located optimally between avalanche, terrain, and blast radius hazards.

Underground productivity is viewed as an opportunity for the Kemess Project. The current production limitation is the capacity of the paste backfill plant. The stope sizing, deposit geometry, and efficiency of the underground design are all viewed as comparable to other globally benchmarked longhole

operations in the 10–12 ktpd range. The underground mining rate will be revisited as further study of the KUG longhole mine progresses.

1.15.3 Metallurgy and Processing

Metallurgical testwork for the Main Zone and KUG deposits shows that a conventional sulphide flotation flowsheet with a primary grind of ~150 µm and regrind of ~20 µm is appropriate for the project. Main Zone composites achieved copper recoveries of 83–90% and gold recoveries of 54–59%, while KUG material demonstrated higher recoveries, particularly for copper. Mineralogical analysis shows chalcopyrite as the dominant copper mineral and pyrite as the primary gangue sulphide, with gold frequently locked within pyrite – supporting the inclusion of a leach circuit to recover gold from cleaner-scavenger tails.

Comminution testing reveals meaningful variability between Broken and Not-Broken Main Zone mineralized material types, with A×b values ranging from 40–69 and Bond BWi values of 16–18 kWh/t, indicating material hardness differences that will influence plant throughput. Additional variability testing is recommended to refine grinding energy estimates and support detailed design.

Deleterious elements in both feed and concentrate are low and well below penalty thresholds, with silica in Broken Zone concentrate being the only parameter requiring attention through blending. Cyanide leach tests on cleaner tails produced 55–76% gold extraction, validating the assumed 70% leach recovery used for PEA modelling.

Collectively, the testwork supports the selected flowsheet, highlights the importance of ore-type-based modelling, and identifies targeted areas for further metallurgical and comminution testing in the next project phase.

1.15.4 Infrastructure

Road upgrades are required north of the Kemess plant site to accommodate the movement of mine haul trucks to and from the Main Zone pit operation. Bridge upgrades are necessary but only for use by tare (unladen) haul trucks.

The loadout facility at Mackenzie will need to be upgraded to be able to simultaneously service the Kemess mine in addition to the Mount Milligan Mine.

The existing personnel camp will be refurbished and additional modules installed to accommodate the anticipated larger workforce.

A new infrastructure pad is required at the Kemess North site to accommodate a truck shop, fuel station, paste backfill plant, emergency station, and offices. Other installations at the North site include a communications network and security facilities.

The paste backfill system requires additional thickener tanks at the mill and a 7 km pipeline to the KUG portal where the paste fill plant and binder storage tank will be constructed. An underground booster system will be constructed in the decline to pump the paste fill through the 3.6 km decline to the underground mine distribution system.

The main substation supplying the transmission line will require upgrades to support power delivery to both the Kemess Project and the Mount Milligan Mine. Engagement with BC Hydro is ongoing to confirm upgrade requirements and to secure sufficient electrical power capacity for the Project.

A new 13.8 kV distribution line will be constructed to supply power to the Kemess North and underground mining areas. The final routing of this line will be determined in the next phase of studies and is expected to generally follow the conveyor alignment.

The KUG TSF and Kemess South TSF are capable of storing the proposed quantity of tailings. The final elevation of the Kemess South TSF will be approximately 1,537 masl with a 50 m extension and raise to 1,417 masl for the current buttress. The KUG facility will be filled to maximum tailings elevation of 1,257 masl.

Stability analyses were completed for the Kemess South TSF embankments at their respective ultimate design elevations. Analyses used geotechnical material parameters from previous stability assessments which were based on historic laboratory testing, site investigations, and construction material testing. All dams were found to meet or exceed the minimum factors of safety outlined by the Canadian Dam Association (CDA) and the Health Safety and Reclamation Code (HSRC) for Mines in British Columbia.

Capital allocations for identified infrastructure installations and upgrades have been included in the Project capital cost estimate.

1.15.5 Environmental and Social

The Kemess Project benefits from existing permits maintained for the idle Kemess South site. With acceptance of the proposed plan of development, early works activities may be initiated to compress the Kemess Project development schedule.

A significant library of environmental knowledge has been accumulated on the Kemess site from many years of monitoring, measurement and compliance reporting. The knowledge, along with defined procedures and ongoing monitoring, will guide future activities. Field work to complete the environmental studies commenced in 2024, with the aim to have baseline collection completed by the end of 2027/early 2028.

The Kemess site possesses an extensive biological database for fisheries and aquatic resources, with continuous data collection dating back to 1992 for sediment quality and fisheries resources. This long-

term monitoring satisfies commitments under the Fisheries Compensation Agreement and the Fish and Aquatic Effects Monitoring Plan.

The proactive and transparent relationship Centerra maintains with its Indigenous partners, Takla, Kwadacha and Tsay Keh Dene First Nations, and Gitxsan Nii Gyap First Nations is highly valued.

1.16 RECOMMENDATIONS

Project recommendations pertain to the work required for a preliminary feasibility study towards a declaration of mineral reserves and preparation for a feasibility study. Recommendations are segmented by technical subject.

1.16.1 Geology, Exploration, and Mineral Resource Estimation

The Kemess deposits are categorized as tilted tabular calc-alkalic copper-gold porphyry deposits. Recent exploration by Centerra has continued to define the Kemess Main Zone mineralization trend, an east-west belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east the trend includes the Nugget zone, KUG deposit, Kemess Offset Zone KOZ), Kemess East deposit, and Hilda South targets. There is a general trend of deposits occurring at greater depths towards the east.

High priority brownfield exploration and infill targets for resource growth are along the western and eastern margins of the updated resource shells, including the Nugget and Kemess Offset zones. The KOZ between KUG and Kemess East should be drilled to identify any significant mineralization that exists in the fault block between the two deposits.

Recent drilling at Kemess East confirmed that the Kemess East deposit remains open to the south and at depth. Further drilling is needed to test the mineral potential of this zone and fully define the extents of the deposit.

The drilling data and updated resource model at Kemess South show potential for additional shallow and deep mineralization west of and below the historical pit. Future drilling is recommended west of the Kemess South open pit in the West Fault area to target potential resource expansion both at surface and at depth.

Additional greenfield exploration work is recommended to advance near surface porphyry targets in the Kemess district and adjacent properties. Recommended additional geophysics surveys include airborne mobile magnetotelluric surveys and detailed ground induced polarization (IP) surveys, regional geochemistry surveys including stream sediment sampling as well as soil and till sampling grids, regional data compilation, and ongoing helicopter supported diamond drilling programs.

Budgets for programs over the next three years should be on par with Centerra programs in recent years. Annually, these budgets have been in the range of CA$4–10 million comprising diamond drilling, geophysics surveys, geochemical sampling, and development of a comprehensive three-dimensional (3D) exploration model and drilling database. Ongoing advancement of the 3D geology and exploration models for both the Kemess deposits and the Kemess greenfield district are recommended.

Specific to the Main Zone and KUG zone, the QP Geologist makes the following recommendations:

- Complete additional infill and step-out drilling to improve definition of fault continuity, orientation, and spatial extent
- Incorporate oriented core drilling and/or televiewer data, where feasible, to improve confidence in fault geometry and kinematic interpretations
- Refine the structural model by integrating 3D fault representations, where supported by data, to better constrain fault thickness and potential impacts on mineralization
- Integrate and reconcile the CSA Global and in-house alteration models for the Kemess Main Zone
- Expand spectral analysis and magnetic susceptibility data coverage to improve alteration classification
- Develop and incorporate an alteration model for Kemess South
- Update the geological and Mineral Resource models as additional drilling, structural, and alteration data become available.

1.16.2 Geotechnical Engineering, Hydrogeology

Due to the interaction between the open pit and underground areas of the Main Zone, detailed design, scheduling and computational modelling is recommended. 3D computational modelling is recommended and should consider multiple milestones according to the mining schedule and projected advance of surface and underground mining. Other recommended geoengineering work is:

- Laboratory testing of representative tailings material for suitability as cement paste backfill
- Additional phases of drilling from surface, associated laboratory testing and analyses to further characterize the rock mass of the Main Zone and KUG
- Hydrogeological studies including permeability testing for the Main Zone as well as piezometers installations in the Main Zone to monitor phreatic levels
- Geohazard assessments of surface infrastructure and access roads are recommended to design appropriate engineering and administrative controls.

The recommended geotechnical and hydrogeological engineering work is projected to cost approximately US$5 million.

1.16.3 Mining

Geotechnical Engineering

Recommended pit slope geotechnical study should focus on the following key action items:

- Additional geomechanical and hydrogeological drilling, sampling and testing to cover the data gap zones of the proposed open pits
- Additional televiewer surveys in select exploration and/or geotechnical drillholes to enhance the rock mass structural database
- Upgrade of the 3D lithology, alteration, oxidization, and fault models, including the locations, continuities, and characteristics of local major structures
- Refinement of the geotechnical model, particularly for the broken zone delineation
- Development of a hydrogeology model for pit dewatering plan
- Enhancement of terrain hazard assessment to include ground truthing and refined stability analyses on the natural slopes surrounding the proposed pits.

Mining

Recommendations for mining pertain to advanced engineering studies towards completion of a prefeasibility study in 2026, including:

- Equipment trade-off studies, and updated manpower estimates. Trade-off studies should include utilization of new technologies such as automation and electrification.
- Collection of equipment price quotations, fabrication times and estimated delivery dates to provide updated capital costs and schedules.
- More detailed planning and scheduling to provide annual estimates of production, revenues and costs.
- Ventilation simulations with updated mine designs, equipment requirements and mining schedules to verify key design, capital and operating cost assumptions.
- Underground and surface material handling simulations to finalize mechanical designs, capital and operating cost projections.

Backfill System

Additional studies are required to validate key assumptions and reduce technical uncertainty, itemized by the following recommendations:

- Paste test work to include material characterization, rheology testing, thickening and filtration testing, and unconfined strength testing
- Surface vs underground paste plant location trade-off study
- Tailings thickener location trade-off study
- Binder study to select optimum and most economic binder solution for the remote location of Kemess
- Cyclone study to develop a targeted particle size distribution (PSD) which may improve filtration and binder consumption.

1.16.4 Metallurgy and Processing

The following activities are recommended to further advance the metallurgical understanding of the deposit and to finalise the process plant design:

- Equipment price quotations/lead time scheduling, reagent estimation, power estimates, training cost estimation
- Complete variability work on SMC, Bond BWi, and Ai testing across Main Zone domains – especially Not-Broken material – to refine throughput, energy demand, and mill sizing assumptions
- Conduct more leach testing across mine-life composites, investigate cyanide-soluble copper impacts, and refine the 70% gold leach recovery assumption with broader datasets
- Undertake more detailed gold liberation and deportment studies, ensuring consistent grind-size reporting and sample naming across all mineralogical datasets
- Complete detailed inspections of the existing equipment in the mill to de-risk future engineering works in the process plant
- Optimize the flotation circuit by completing trade off studies for flotation technology and overall circuit layout.

1.16.5 Infrastructure

Significant infrastructure exists at Kemess. Uncertainties to be investigated during the prefeasibility study include:

- Power supply and substation requirements of the BC Hydro grid.

- Engineering and construction cost estimates for the existing and proposed personnel camp expansion.
- Trade-off studies are recommended to optimize infrastructure locations. Consideration should be made to minimize exposure to geological and other hazards. Additionally, infrastructure siting should consider weather conditions, serviceability and constructability.
- Budget quotations for critical infrastructure, first-fills and critical components should be requested from reputable manufacturers.

Tailings Infrastructure

The following activities are recommended as part of the ongoing design of the KUG and Kemess South TSFs:

- Conduct testwork to establish composition and strength parameter of tailings. This study has been based on tailings testwork completed as part of the KUG development (Golder, 2019) which may vary from the Nugget and Main Dam Pit tailings. Testwork should include:
  - index testing
  - settling and consolidation analysis
  - chemical testing to confirm rougher tailings assessment of whether they are acid-generating and/or metal leaching
  - material strength testing, specific proposed cyclone sand material strength testing.
- Site investigation, detailed characterization, and subsurface material interpretation for the diversion dams and North Saddle Dam are required to refine the design. Currently, minimal data is available in these locations, and with updated designs larger than the initial footprints, additional areas need to be investigated.
- Detailed stability and sensitivity assessments are required to evaluate dam structural performance using refined input parameters based on confirmed embankment materials and laboratory testing. Detailed seepage analyses and modelling should be conducted for all embankments utilizing updated material properties to estimate the phreatic surface and evaluate potential impacts on downstream water management infrastructure.
- More detailed design of access roads, diversions, and closure spillway.
- Complete a water balance model that reflect the tailings management strategy within both facilities.
- An updated deposition model should be developed, incorporating revised pond sizing and a defined tailings surface. This should consider minimum tailings beach length of 500 m, a

minimum of 2 m water depth below the barge for adequate water reclaim, and maintenance of the pond in the optimal location within the facility.

- Assessment of the existing water management structures including the Kemess South TSF Seepage Recycling Pond and Sediment Pond to determine current capacity and if they accommodate TSF raises.
- Establish the cyclone sand placement methodology.
- Review existing site investigation data and perform additional site investigation to characterize foundation conditions, material strength parameters, specifically under the South Diversion Dam, North Diversion Dam, proposed East Diversion Dam, purposed North Dam area, roads, spillways where limited data exists.
- Further optimization of diversion and water management design and associated diversion structures.
- Further definition of tailings geotechnical and geochemical characteristics.
- Detail tailings deposition to define staged embankment construction and staged tailings deposition.
- Updated stability assessments incorporating refined input parameters for cyclone sand, based on further review of existing material properties and finalized tailings splits.
- Preliminary engineering designs for supporting infrastructure, including, access roads, tailings pipelines, cyclone facilities, water management ponds and systems.
- Water quality modelling to support closure plans and project cost.

1.16.6 Environment and Social

Recommended tasks for environmental stewardship and social interactions include:

- Additional studies to assess new areas to be disturbed such as: new mine waste storage facilities, the conversion of the existing permitted block cave subsidence zone to an open pit configuration, new conveyor alignment, infrastructure, etc. This will include, but not be limited to, additional studies geochemical characterization and water quality and quantity, fisheries and aquatic resources, archaeology, soils, vegetation, and wildlife.
- Continue and finalize field work to complete the environmental studies commenced in 2024. with the aim to have baseline collection completed by the end of 2027/early 2028.
- Advance studies and testwork to support the permitting of the leach plant
- Complete a Site Performance Objective (SPO) to establish long-term selenium permit limits for Waste Rock Creek, supported by ongoing studies such as the Bird and Amphibian

Selenium Bioaccumulation study initiated in early 2025 to manage selenium inputs from a previously operated WRSF in the Kemess South Mine Area.

- Continue discussions with affected First Nations to maintain Project transparency and support.

2 INTRODUCTION

This Technical Report, which was prepared by and for Centerra, has been prepared under the supervision of Mr. Christopher Richings, P.Eng., and Vice-president of Technical Services for Centerra Gold, in accordance with the disclosure and reporting requirements set forth within Canadian Securities Administrators National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101), Companion Policy 43- 101CP, and Form 43-101F1, as amended. Mr. Richings, Ms. Sica, and Mr. Rowe are qualified persons under NI 43-101. The QPs are employees of Centerra.

The Technical Report summarizes the Mineral Resources for the Kemess Restart Project ('Kemess', 'the Property', 'the Project' or 'KRP') and the results of a preliminary economic assessment (PEA) study to establish a new mining area and restart the existing process plant.

The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

In January 2018, Centerra acquired its 100% interest in AuRico which owned a number of assets including the idle Kemess Mine.

Kemess is located in remote north-central, British Columbia, Canada, approximately 325 km north-northwest of Fort St. James and 200 km northeast of Smithers. This Technical Report summarizes a PEA completed in 2025 on the Project, incorporating changes to the mineral inventory and Project Plan as compared to the prior Technical Report on Kemess filed in 2016 by AuRico.

Centerra, a global mining company organized under the laws of Canada, is engaged in the acquisition, exploration, development, and operation of mineral properties. Shares of Centerra are listed on the Toronto Stock Exchange under the trading symbol "CG" and the New York Stock Exchange under the trading symbol "CGAU".

All dollar figures in this Technical Report refer to US Dollars (US$), unless otherwise noted.

2.1 SOURCES OF INFORMATION

This Technical Report is based on published material and data, professional opinions, and unpublished materials available to Centerra or prepared by its employees and consultants. In addition, certain information used to support this Technical Report was derived from previous technical reports on the Project and from reports and documents listed in Item 27 (References). Other sources of data include

geologic and block model reports, drill hole assay data, the block model, mine plans, cost estimates, and economic models that were prepared by employees of Centerra.

2.2 CONTRIBUTING PERSONS AND SITE INSPECTIONS

This Technical Report has been prepared by the persons listed in Table 2-1, each of whom is a Qualified Person (QP), as defined by NI 43-101, and has provided a QP certificate.

Other Centerra employees compiled certain Items of this Technical Report under the supervision of those identified in Table 2-1. These Centerra employees are experienced technical and accounting/finance professionals in their respective areas of expertise.

Table 2-1: QPs and responsibilities

Qualified Person	Title	Primary area(s) of responsibility
Christopher Richings	VP Technical Services	Items 2–5, 14, 15, 16, 19, 20, 22-24; parts of Items 1,12, 21, 25, 26
Cheyenne Sica	Director, Exploration Canada	Items 6–11; parts of Items 1, 12, 25, 26
Gerard Rowe	Director, Engineering	Items 13, 17, 18; parts of Items 1, 12, 21, 25, 26

Standard professional procedures have been followed in preparing the contents of this Technical Report. Data used in this Technical Report have been verified, where possible, and all data is considered to have been collected in a professional manner.

Site inspections by QPs are detailed in the QP certificates earlier in this Technical Report.

2.3 UNITS OF MEASURE

This Technical Report utilizes metric units throughout as set forth in the Glossary included in Item 28. Grades are in percent of copper metal by weight and grams per tonne (g/t) for gold and silver. Masses are expressed in metric tonnes of 2,204.6 pounds. Estimated gold and silver production and sales are measured in units of Troy ounces with a conversion of 31.1035 grams per Troy ounce.

3 RELIANCE ON OTHER EXPERTS

All technical information supplied by external consultants to support this report was reviewed and accepted by the named QPs in Item 2.2.

The QPs have relied upon information and opinions provided by Centerra employees with relevant experience in legal, tax, royalty, environmental, and permitting matters. The QPs consider the sources of information to be reliable and appropriately qualified.

4 PROPERTY DESCRIPTION AND LOCATION

Kemess is located in remote north-central, British Columbia, Canada, approximately 325 km north-northwest of Fort St. James and 200 km northeast of Smithers. Access to the Kemess property is via the all-season Omineca resource road from the town of Mackenzie.

The Kemess property is located in the northern and central plateaus and mountainous physiographic region of British Columbia. This region is dominated by flat to rolling topography with mature erosional surfaces.

4.1 PROPERTY LOCATION

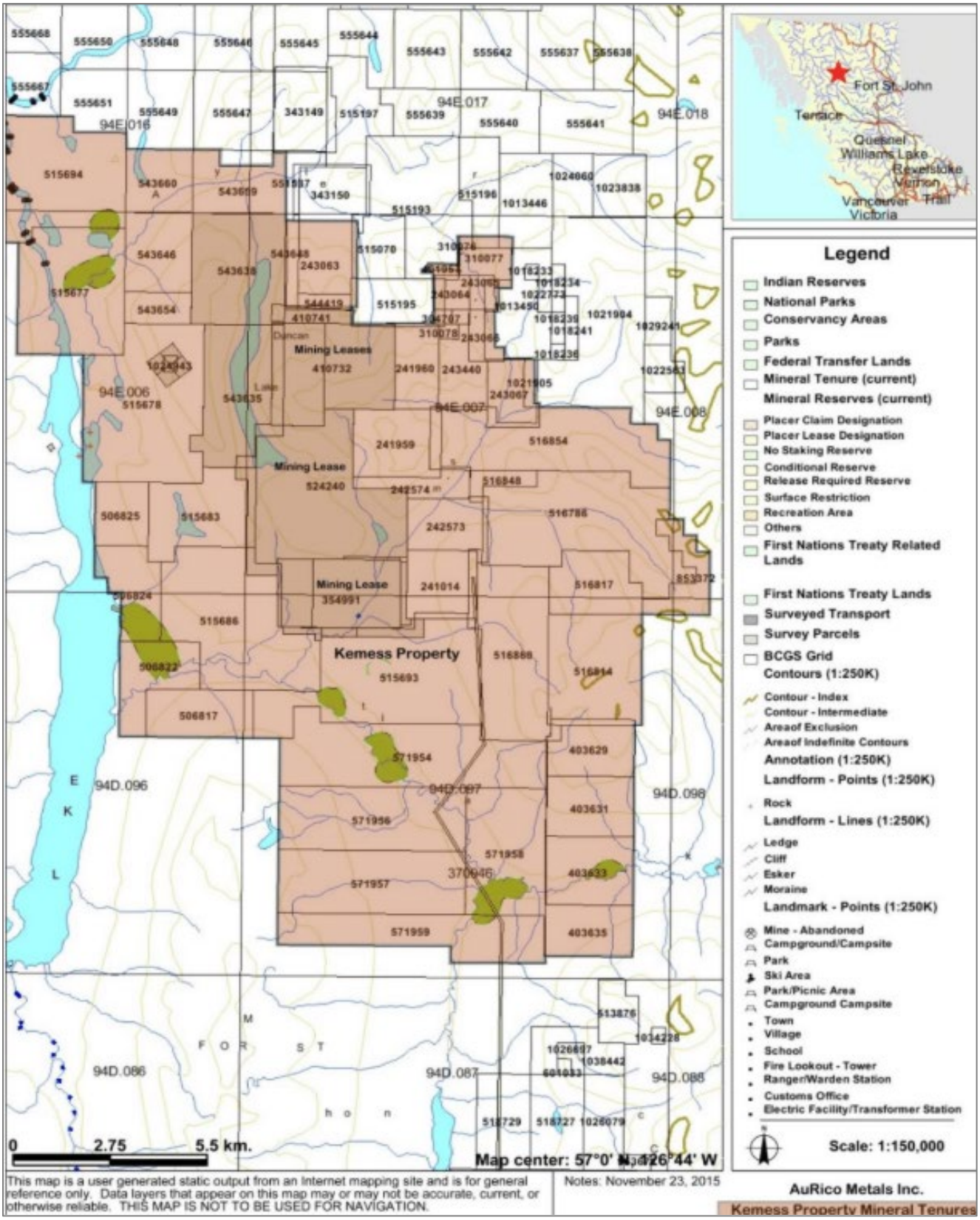
The Kemess property is located approximately 250 km north of Smithers and 430 km northwest of Prince George at 57°02’ north latitude and 126°47’ west longitude in the mountainous area east of the Spatsizi Plateau and west of the Swannell Ranges near Thutade Lake (Figure 4-1). The property spans the boundary between the 94E and 94D NTS sheets and is within the Omenica Mining Division. Elevations range from 1,200 m to 2,000 m, with the tree line occurring at approximately 1,500 m.

Figure 4-1: Kemess Project location

The property spans the boundary between the 94E and 94D NTS sheets and is within the Omenica Mining Division.

AuRico holds mineral title to 53 claims totaling 29,117.56 ha. AuRico also has leasehold on an additional four claims totalling 3,483.33 ha. AuRico agreed to forfeit 9 mineral claims (6,774.08 ha) in the 2017 Impact Benefit Agreement (IBA) (Figure 4-2).

Figure 4-2: Kemess mineral claims



The Property is comprised of 58 mineral claims covering an area of 29,302.7 ha, the details of which are provided in Figure 4-3.

Figure 4-3: Mineral tenure information as recorded online with BC Mineral Titles

AuRico Metals Inc. - Kemess Property, BC

Summary

	No.	Hectares	Acres
A. Mining Leases	4	3,483.330	8,607.308
B. Mineral Claims	58	29,302.692	72,406.951
	62	32,786.022	81,014.260

AuRico Metals Inc. - Client No. 281385

Mining Leases

Tenure No.	Name/District Lot	Owner	Map No.	Lease Term Date	Good To Date	Area (ha)
354991	Kemess South Lease: DL's 7198, 7199, 7200, 7201,7204, 7207	281385 (100%)	094E007	2027/sep/15	2016/sep/15	862.330
410732	Kemess North Lease: DL's 7032, 7328	281385 (100%)	094E007	2034/sep/29	2016/sep/29	950.000
410741	Kemess North Lease: DL.7329	281385 (100%)	094E007	2034/sep/29	2016/sep/29	106.000
524240	Midway Lease: DL.7342	281385 (100%)	094E	2035/dec/22	2016/dec/22	1565.000
4 Mining Leases					**Hectares:**	**3483.330**

Mineral Claims

Tenure No.	Claim Name	Owner	Map No.	Issue Date	Good To Date	Area (ha)
241014	SEM #1	281385 (100%)	094E007	1989/jul/18	2025/dec/14	400.000
241959	NEK 3	281385 (100%)	094E007	1990/may/03	2025/dec/14	500.000
241960	NEW KEMESS 3	281385 (100%)	094E007	1990/may/03	2025/dec/14	375.000
242573	DU 2	281385 (100%)	094E007	1990/aug/02	2025/dec/14	500.000
242574	NEK 4	281385 (100%)	094E007	1990/aug/01	2025/dec/14	350.000
243063	CAN 1	281385 (100%)	094E007	1991/feb/02	2025/dec/14	500.000
243064	DUNC 1	281385 (100%)	094E007	1991/feb/01	2025/dec/14	100.000
243065	DUNC 2	281385 (100%)	094E007	1991/feb/01	2025/dec/14	100.000
243066	DUNC 3	281385 (100%)	094E007	1991/feb/01	2025/dec/14	150.000
243067	CREEK	281385 (100%)	094E007	1991/feb/02	2025/dec/14	300.000
243440	ALISON 1	281385 (100%)	094E007	1991/may/14	2025/dec/14	500.000
304706	GOZ 1	281385 (100%)	094E007	1991/sep/21	2025/dec/14	25.000
304707	GOZ 2	281385 (100%)	094E007	1991/sep/21	2025/dec/14	25.000
310076	DUN 1	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
310077	DUN 2	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
310078	DUN 3	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
355408	MILL CREEK 4	281385 (100%)	094E007	1997/apr/23	2025/dec/14	25.000
401957	UN 1	281385 (100%)	094E007	2003/apr/29	2025/dec/14	50.000
403629	BEAR 6	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000
403631	BEAR 8	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000
403633	BEAR 12	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000

Tenure No.	Claim Name	Owner	Map No.	Issue Date	Good To Date	Area (ha)
403635	BEAR 16	281385 (100%)	094E007	2003/jul/09	2025/dec/14	375.000
405949	LAT 1	281385 (100%)	094E007	2003/oct/05	2025/dec/14	25.000
414229	DUNC 4	281385 (100%)	094E007	2004/sep/15	2025/dec/14	25.000
414230	DUNC 5	281385 (100%)	094E007	2004/sep/15	2025/dec/14	25.000
414231	UN 2	281385 (100%)	094E007	2004/sep/16	2025/dec/14	25.000
414232	UN 3	281385 (100%)	094E007	2004/sep/16	2025/dec/14	25.000
506817	TLK 1	281385 (100%)	094D	2005/feb/11	2025/dec/14	423.346
506822	TLK 2	281385 (100%)	094D	2005/feb/11	2025/dec/14	423.244
506824	TLK 3	281385 (100%)	094D	2005/feb/11	2025/dec/14	387.724
506825	TLK 4	281385 (100%)	094E	2005/feb/11	2025/dec/14	281.830
515677		281385 (100%)	094E	2005/jun/30	2025/dec/14	1108.035
515678		281385 (100%)	094E	2005/jun/30	2025/dec/14	1443.310
515683		281385 (100%)	094E	2005/jun/30	2025/dec/14	669.335
515686		281385 (100%)	094D	2005/jun/30	2025/dec/14	1427.856
515693		281385 (100%)	094D	2005/jun/30	2025/dec/14	1534.100
515694		281385 (100%)	094E	2005/jun/30	2025/dec/14	1353.164
516786		281385 (100%)	094E	2005/jul/11	2025/dec/14	1391.637
516814		281385 (100%)	094D	2005/jul/11	2025/dec/14	863.906
516817		281385 (100%)	094D	2005/jul/11	2025/dec/14	440.555
516848		281385 (100%)	094E	2005/jul/11	2025/dec/14	105.661
516854		281385 (100%)	094E	2005/jul/11	2025/dec/14	1197.161
516860		281385 (100%)	094D	2005/jul/11	2025/dec/14	1075.379
543635		281385 (100%)	094E	2006/oct/19	2025/dec/14	897.552
543638		281385 (100%)	094E	2006/oct/19	2025/dec/14	861.624
543646		281385 (100%)	094E	2006/oct/19	2025/dec/14	439.571
543648		281385 (100%)	094E	2006/oct/19	2025/dec/14	105.501
543654		281385 (100%)	094E	2006/oct/19	2025/dec/14	175.891
543659		281385 (100%)	094E	2006/oct/19	2025/dec/14	421.784
543660		281385 (100%)	094E	2006/oct/19	2025/dec/14	351.483
544419		281385 (100%)	094E	2006/oct/25	2025/dec/14	70.351
571954		281385 (100%)	094D	2007/dec/14	2025/dec/14	1340.848
571956		281385 (100%)	094D	2007/dec/14	2025/dec/14	988.435
571957		281385 (100%)	094D	2007/dec/14	2025/dec/14	988.868
571958		281385 (100%)	094D	2007/dec/14	2025/dec/14	847.371
571959		281385 (100%)	094D	2007/dec/14	2025/dec/14	830.261
853372	Kemess Creek 1	281385 (100%)	093D & 094E	2011/may/03	2025/dec/14	264.315
1024943	CAIRNX	281385 (100%)	094E	2014/jan/09	2026/jan/09	17.596
58 Mineral Claims					**Hectares:**	**29302.692**

The Project site comprises an open pit mine, underground mine, waste rock storage facility (WRSF), tailings storage facility (TSF), mineralized stockpiles, a processing plant, workshop, warehouse, administration buildings, and camp. Figure 4-4 provides a plan view of the Project.

Figure 4-4: Aerial view of Kemess Project site with project extent overlay

4.2 PROPERTY DESCRIPTION

Broad, open, drift, and moraine covered valleys characterize the area, which yield to sub-alpine plateaus and rugged, incised peaks and cirques. Elevations range from 1,200 m to 2,000 m, with the alpine tree line occurring at approximately 1,500 m.

All the work completed during the 2002–2010 drill programs occurred above the tree line in three cirques that open to the north, forming a common southern headwall. Lower elevations on the property are moderately vegetated with spruce-willow-birch forest, while poorly drained areas form peat bogs populated by alder brush, willow, and stunted spruce trees.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 ACCESS

Access to the Kemess property is via the all-season Omineca resource road from the town of Mackenzie (Figure 5-1). Road distance to the site of the Project is 340 km from MacKenzie (population approximately 3,275) and 500 km from Prince George (population approximately 80,000).

Figure 5-1: Road access to Kemess Project

Previous operations at Kemess South were serviced by scheduled year-round flights from various locations in British Columbia. The Kemess Main Zone and Kemess Underground (KUG) deposit lie approximately 6.5 km to the north of the existing Kemess South processing plant and other infrastructure. The deposit is located beneath two north facing alpine cirques with ground surface elevations ranging from 1,500 m to 2,000 m, all above the alpine tree line. Vehicular access to the area overlying the deposit is currently limited to summer months only. Helicopter access is possible during winter. The Kemess South processing plant and other infrastructure are located at an elevation of approximately 1,200 m and are accessible year-round. The Kemess East deposit is located approximately 1 km east of the KUG deposit.

Below the tree line, broad, open, drift, and moraine covered valleys characterize the area. These areas are moderately vegetated with spruce-willow-birch forest, while poorly drained areas form peat bogs populated by alder brush, willow, and stunted spruce trees.

The Project would operate on a fly-in/fly-out (FIFO) basis with rotating rosters ensuring uninterrupted, continuous mining and processing operations.

5.2 CLIMATE

The area climate is generally moderate with short, cool summers and cold winters, although snow can occur during any month. Temperatures range from –35°C to 30°C and average annual precipitation amounts to 890 mm. Extreme weather conditions are possible at the higher elevations.

Mining and processing activities can occur throughout the year.

5.3 LOCAL RESOURCES

Labour and services are available from the surrounding towns of Prince George, Fort St. James, Mackenzie, Vanderhoof, Smithers and Fraser Lake. Workers will reside in bunkhouse accommodations installed by Centerra.

5.4 REGIONAL INFRASTRUCTURE

Infrastructure available to the Project comprises the main forest service roads accessing the Property from the south. Electric power is available on site, delivered by BC Hydro from the BC Hydro Kennedy Substation, south of Mackenzie. Canadian National Railway service is available from Fort St. James and Mackenzie, which connects to the major western and eastern rail routes. Refer to Item 18 for additional information regarding Project infrastructure.

5.5 PHYSIOGRAPHY

The Kemess property is located in the northern and central plateaus and mountainous physiographic region of British Columbia. This region is dominated by flat to rolling topography with mature erosional surfaces. The mountain chains tend to exhibit lower relief than the coastal and southeastern mountains and glacial drift can be quite thick.

6 HISTORY

6.1 PROPERTY OWNERSHIP

Kennco initially staked the northern Kemess ground in 1966 following up on regional silt geochemical surveys, and continued exploration on the property until 1971. Other early operators included Getty Mines Ltd and Shell Oil who optioned the property from 1975 to 1976.

El Condor Resources optioned the property from Kennco in 1986 and conducted exploration through 1992. During this period the discovery hole was made at Kemess South, and the first resource was published on the Kemess North deposit, later named the Kemess Underground (KUG) deposit.

Royal Oak Mines acquired the property in 1996 and went on to develop the Kemess South operation in 1998. Northgate Minerals Exploration acquired the Kemess South deposit and associated Kemess North property in 2000 from Royal Oak Mines. Northgate was subsequently acquired by AuRico Gold in 2011. AuRico Gold and Alamos Gold merged in 2015 and AuRico Metals Inc. was spun off from that merger, with the Kemess asset allocated to AuRico Metals.

Centerra announced the acquisition of AuRico Metals Inc. in November 2017, which included 100% ownership of the Kemess property. The acquisition closed on January 19, 2018.

In May 2018, Centerra sold 100% of the silver production from the Kemess project to a subsidiary of Triple Flag Mining Finance Ltd in exchange for cash sideration of US$45 million as an advance payment, payable in tranches of US$10 million, US$10 million, US$12.5 million and US$12.5 million on the public announcement by Centerra that its board of directors has approved a construction decision with respect to the Kemess project and the three succeeding anniversaries of such date, respectively. In addition, Triple Flag will make ongoing payments of 10% of the then current market price for each ounce of silver delivered.

Table 6-1 summarizes historical ownership, mineral exploration and development activities of the area that has become the Kemess mine and property in British Columbia, (modified from SRK Consulting (Canada) Inc., 2016(SRK)).

Table 6-1: Kemess historical ownership, exploration, and development timeline

Period	Company	Work completed
1966–1971	Kennco Explorations Ltd	Staked 100 two-post mineral claims, regional stream and soil geochemistry, mapping at 1:9,600 scale and completed 232 m of x-ray core drilling in 8 holes.
1975–1976	Getty Mines Ltd and Shell Oil	Optioned property from Kenneco and completed 1:4,800 scale mapping, orthomapping, re-staking, geochemical surveying, and 2,065 m of diamond drilling in 13 holes (75–18 to 75–30). Option dropped in 1977.

Period	Company	Work completed
1986–1992	El Condor Resources Ltd	In 1986, El Condor optioned the property from Kennco and commenced sustained exploration that resulted in the discovery at Kemess South. Over a six-year period at Kemess North, El Condor collected 1,025 rock samples and 5,402 soil samples, completed 76.85 km of IP work, and drilled 14,328 m of core in 69 holes. Additional work included 167 km of line cutting, 54.5 km of roads, and 475 m of trenching. A resource of 157 Mt at 0.37 g/t Au and 0.18% Cu at Kemess North.
2000	Northgate Exploration Ltd	Completed 4,104 m of diamond drilling in 12 holes identified a new higher-grade porphyry zone located east of El Condor's discovery. This work increased the resource at Kemess North to 360 Mt at 0.299 g/t Au and 0.154% Cu.
2001	Northgate Exploration Ltd	Completed 8,220 m of diamond drilling in 16 holes, which increased resources to 442 Mt at 0.40 g/t Au and 0.23% Cu.
2002	Northgate Exploration Ltd	In February 2002, Northgate filed a technical report to the Canadian Securities Regulatory Authorities that included a mineral resource statement for the contemplated open pit deposit. Completed 33,686 m of diamond drilling in 58 holes (41 holes on Kemess North, 5 holes on Kemess East, and 12 holes at Nugget).
2003	Northgate Exploration Ltd	Completed 21,851 m of diamond drilling in 61 holes (24 holes for exploration, 24 holes for condemnation, 7 holes for geotech, and six holes for water monitoring).
2004	Northgate Minerals Corp.	Completed 9,970 m of diamond drilling in 32 holes (16 holes on Kemess North and Nugget, 5 condemnation holes, 4 holes on Duncan Ridge, 6 holes at Hilda, and one hole at Kemess Centre). In June 2004, following further drilling, Northgate published a revised mineral resource estimate, which formed the basis of a prefeasibility study.
2005	Northgate Minerals Corp.	In January 2005, Northgate completed the Kemess North Feasibility Study together with updated mineral reserve and resource estimates. The feasibility study envisaged a large open pit mining operation with tailings from an expanded milling operation deposited in the nearby Amazay Lake (Duncan Lake). In May 2005, Northgate filed a NI 43-101 technical report titled "Revised Mineral Reserve and Resource Kemess North Project". However, Northgate subsequently was unable to obtain regulatory approval to develop the deposit in the manner envisaged in the feasibility study. Completed 16,158 m of diamond drilling in 40 holes (3 holes on Kemess North, 4 holes on NOR1, 3 holes on Kemess East, 3 holes on the Kemess North Offset Zone, 18 holes on the Bear Claims, 1 hole on the Nugget Zone, 2 holes on the Orion Zone and six holes at Duncan Ridge). Hole KN-05-24 discovered the Kemess North Offset Zone.
2006	Northgate Minerals Corp.	Completed 8,689 m of diamond drilling in 18 holes (9 holes on Kemess North, and 9 holes on Kemess East). Completed Titan 24 IP survey along length of KN trend. Hole KH-06-03 discovers Altus zone at Kemess East.
2007	Northgate Minerals Corp.	Completed 18,132 m of diamond drilling in 28 holes (three holes on Kemess North, 24 holes on Kemess East and one hole on NOR1). Completed Titan 24 IP survey grid at Kemess East. Hole KH-07-04 discovers Ora zone at Kemess East.
2008	Northgate Minerals Corp.	22.6 line-km of IP survey and mapping over the Creek claims southeast of Kemess East, directly north of the Kemess South TSF to follow up on grab samples collected during the 2007 exploration season.
2010	Northgate Minerals Corp.	In May 2010, Associated Mining Consultants Inc. (AMC) completed a review of the deposit. The review concluded the potential to mine the historical Kemess North (KUG) by block caving methods. Completed 16,439 m of diamond drilling in 30 holes on Kemess Underground deposit. targeting the potential mining area.
2011	Northgate Minerals Corp.	In February 2011, released a mineral resource estimate on the KUG deposit. In August 2011, Northgate announced the results of a PEA for the KUG project which outlined a cumulative production of 1.1 Moz of gold and 490 Mlb of copper over a mine life of approximately 12 years.

Period	Company	Work completed
2011	AuRico Gold Inc.	Completed 2,207 m of diamond drilling in 3 holes on KUG deposit and 3,962 m of diamond drilling in 16 holes for geotechnical and hydro-geological purposes in the vicinity of proposed underground infrastructure.
2013	AuRico Gold Inc.	In 2013, AuRico and SRK compiled the feasibility study and technical report on the KUG project with then current long-term metal prices and costs. Completed 13,330 m of diamond drilling in 9 holes (7 holes on Kemess East and 2 holes on the KOZ). Completed regional soil sampling.
2014	AuRico Gold Inc.	Completed 16,872.5 m of diamond drilling in 12 holes on Kemess East. Completed a ZTEM property-wide geophysical survey. Conducted regional soil sampling.
2015	AuRico Metals Inc.	Completed 29,111 m of diamond drilling in 34 holes (12 holes on Kemess East, 3 holes on the KOZ, 9 holes testing regional targets and 10 holes conducting geotechnical drilling for the KUG Feasibility Study). Completed a property-wide LiDAR survey. Conducted regional soil and sampling and mapping.
2016	AuRico Metals Inc.	New technical report – update to the 2013 KUG feasibility study and technical reports based on an updated mineral resource estimate, mine design, infrastructure design and cost estimates. 18,544 m in 13 drill holes at Kemess East to test gaps in the resource model, expand the extent of the deposit and more accurately locate key faults.
2017	AuRico Metals Inc.	An updated resource estimate for Kemess East was released in January 2017 13,930.1 m in 10 drill holes with eight targeting Kemess East and two targeting KOZ to test for extension of mineralization and convert the resource classification.

6.2 HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES

6.2.1 Kemess North (Main Zone)

The Kemess Project and surrounding area has seen mineral exploration since the discovery of placer gold at the mouth of the McConnell Creek in 1889. In the past 50+ years, at least 40 different companies have conducted various exploration activities overlapping the area covered by the current Kemess claim boundary. The first exploration work recorded on Kemess proper was soil and stream geochemistry and mapping by Kenneco starting in 1966, focused around the Kemess North area, including the Main Zone (which includes Nugget), KUG, and Kemess East zones. Exploration continued in 1968 by Cominco Ltd, focused on the Rat and Kemess South areas.

Early exploration work at Kemess through the 1960s and 1970s involved stream, soil and rock sampling geochemistry programs; ortho-mapping as well as 1:9,600 and 1:4,800 scale mapping campaigns; geophysical surveys including induced polarization (IP), magnetic, aeromagnetic, VLF-EM; and 21 diamond drill holes. No further work was conducted at the Kemess North area until the late 1980s.

After the discovery of the Kemess South deposit by El Condor Resources Ltd, an extensive exploration program at the Main Zone was undertaken. Between 1986 to 1994, the company collected 1,025 rock samples and 5,402 geochemical samples; completed 76.85 line-km of ground electromagnetics, 14.1 line-km of ground magnetic surveying, 161.4 line-km of IP, and drilled 14,328 m of core from 69 holes (Golder Associates Ltd, 2017).

After the initial exploration, Kemess Main Zone advanced to a pre-construction mine development project that had feasibility studies completed in 2004–2005 during its consideration as an open pit project called Kemess North. Early exploration work in the area had identified a porphyry target but significant copper and gold grades were not intersected until a deep drilling program in 2001. Following this, exploration focused on expanding the resource base for the envisioned open pit mine. In 2010, detailed review began for an underground project and exploration drilling continued in support of an underground mine model. After a positive feasibility study on the KUG deposit in 2013, AuRico restarted exploration and drilling programs at Kemess.

Exploration at the Main Zone resumed in 2000 with Northgate Exploration who continued to explore and drill the property until 2011 (Table 6-1). Drill programs increased the resources and took the company through a prefeasibility study in 2005. In the same year the discovery hole at the KOZ was drilled, immediately east of the KUG deposit. The discovery led to subsequent geophysical surveys and drilling programs targeting deep high-grade mineralization, and successfully outlined the Kemess East deposit, 1 km east of the KUG deposit along the same inferred mineralization trend. The first mineral resource estimate for the Kemess East deposit was later published in January 2015, by AuRico (Golder Associates Ltd, 2017).

In August 2011, Northgate announced the results of a PEA for the KUG project which outlined a cumulative production of 1.1 Moz of gold and 490 Mlb of copper over a mine life of approximately 12 years. In 2011, AuRico Gold Inc. (AuRico) purchased all assets held by Northgate Minerals Corp. In the same year, AuRico completed additional hydrogeological and geotechnical drilling to advance the KUG project.

AuRico continued exploration and resource definition diamond drilling at Kemess East, and KUG as well as exploration drilling at the Offset Zone, completing over 60,000 m of drilling from 2015 to 2017. A feasibility study for the KUG deposit was completed and released in 2016 by AuRico, as an update to the published 2013 KUG feasibility study. The updated study contemplates the development of a low-cost panel caving operation to extract the KUG reserves over a 12-year mine life (SRK Consulting (Canada) Inc., 2016).

The 2016 AuRico exploration program spanned from June 18 to October 14 and totalled 18,555.5 m in 13 drill holes targeting the Kemess East deposit. The 2016 program successfully completed its primary goals, which were to: 1) test and fill internal gaps in the resource from previous drilling; 2) more accurately define the fault features that bound Kemess East deposit; 3) expand the Kemess East resource by completing step out drilling.

The 2017 AuRico exploration drill program spanned from June 26 to October 30. Drilling totalled 13,930.1 m in 10 drill holes, 8 targeting the Kemess East deposit and two located in the KOZ. Exploration

drilling aimed to test the northern and southern limits of the Kemess East deposit outside of the current resource and to test for additional mineralization in the KOZ to the north of intercepts identified during the 2015 drilling campaign.

In January 2017, AuRico released an updated resource estimate for Kemess East. In January 2018, Centerra Gold Inc. acquired AuRico Metals Inc., which included 100% ownership of the Kemess property. At that time, Kemess was viewed as a brownfield development asset host to the feasibility-stage KUG and PEA-level Kemess East projects.

6.2.2 Kemess South

Pacific Ridge Resources Ltd (Pacific Ridge) staked the area of the Kemess South deposit in 1983. Exploration programs were subsequently carried out by Pacific Ridge and Anaconda Canada Ltd (Anaconda) in 1984; St. Philips Resources Inc. (St. Philips) in 1988 and the Kemess South Joint Venture (JV) between El Condor Resources Ltd (El Condor) (60%), operator, and St. Philips Resources Ltd (40%) from 1990 to 1993. In 1991, Rio Algom Explorations Inc. (Rio Algom) acquired claims adjoining the west and south sides of the Kemess South JV claim holdings (Skrecky, 2007).

The initial work on the property by Pacific Ridge and Anaconda consisted of a limited diamond drilling program to test a gold-copper-molybdenum soil geochemical anomaly. This drilling identified porphyry style gold-copper-molybdenum mineralization, but grades were considered too low and the property was dropped. St. Philips carried out IP surveys, geochemical surveys and reverse circulation drilling, which marginally expanded the mineralized area. The Kemess South (JV completed a major delineation diamond drilling program and exploration activities, including IP and geochemical surveys. In 1992, Rio Algom drilled five holes totalling 1,745 m to further delineate the deeply buried western extension of the Kemess South deposit. In late 1993, the Kemess South JV acquired the claims held by Rio Algom. By the end of 1993, a total of 26,314 m of diamond drilling in 156 holes had outlined a substantial gold-copper deposit that was amenable to open pit development (Skrecky, 2007).

In 1994, the Kemess South JV conducted a nine-hole, 1,867 m infilling drilling program. In 1996, Royal Oak Mines Inc. (Royal Oak) acquired the Kemess South property and drilled 22 shallow due diligence holes totalling 3,316 m. In 1998 Royal Oak commenced operations from the Kemess South ore body. These operations went into receivership in 1999. In 2000, Northgate Exploration Ltd (Northgate) bought the property out of receivership and continued exploration and drilling programs at Kemess South until 2010. Forty-two holes drilled in 2004 were for exploration and definition purposes. These holes were completed under the supervision of Kemess Mines staff. The program was multi purpose. Three areas of the Kemess South pit were explored, including the hypogene mineralization and Takla Group volcanics in the southwest quadrant of the pit, hypogene mineralization in the southeast quadrant of the pit, and a native copper showing in the Toodoggone Formation in the northwest quadrant of the pit. The

2006 drilling programs were spurred by rising copper prices and revisited the east end mineralization as well exploration drilling west of the pit, and geotechnical drilling (Skrecky, 2007).

In May 2007, Northgate released an updated Technical Report with updated resource and reserve statements on Kemess South, and a re-evaluation of the then inactive East Pit. The 2007 report is the last known published Technical Report released on Kemess South (Skrecky, 2007).

6.3 HISTORICAL PRODUCTION

The only successfully exploited porphyry deposit on the Property has been the Kemess South deposit, which produced 2.975 Moz of gold and 749 Mlb of copper from 218 Mt of ore over its 13-year lifespan as an open pit mine (SRK Consulting (Canada) Inc., 2016). There is no historical production from the Main Zone, KUG deposit or the Kemess East deposit, both in the northern part of the Kemess property.

An open pit resource was outlined at the Kemess South deposit, initially discovered by El Condor in the late 1980s. Royal Oak, then owners of the property, commenced operations from the Kemess South deposit in 1998. The operation went into receivership in 1999, and in 2000 Northgate a predecessor company to AuRico Metals, bought the property out of receivership.

Production ceased in 2011, and since then work at Kemess South has been focused on reclamation and site rehabilitation. Historical production from Kemess South since the original start-up is shown in Table 6-2 (from SRK Consulting (Canada) Inc., 2016).

Table 6-2: Historical production of Kemess South Mine

Operator	Year	Waste mined (t)	Ore milled (t)	Grades mill head		Metal produced	
				% Cu	g/t Au	Cu (t)	Au (oz)
Royal Oak	1997	6,014,000	0	0.000	0.000	0.000	0.000
	1998	24,838,324	7,482,909	0.220	0.557	9,687	69,804
	1999	8,668,980	14,113,460	0.212	0.644	21,389	213,793
Northgate	2000	19,911,880	14,089,000	0.222	0.779	23,151	225,998
	2001	17,246,162	15,366,500	0.251	0.855	30,076	277,106
	2002	27,123,742	17,308,300	0.236	0.724	33,051	282,255
	2003	34,617,235	18,633,000	0.225	0.702	34,554	294,117
	2004	36,647,429	18,589,000	0.231	0.735	35,513	303,475
	2005	31,718,631	19,168,000	0.219	0.641	33,440	279,962
	2006	25,502,552	18,233,978	0.244	0.763	36,837	310,298
	2007	24,959,000	17,802,317	0.213	0.627	30,904	245,631
	2008	14,408,998	16,924,271	0.175	0.506	23,549	185,180
	2009	10,259,364	18,352,153	0.160	0.440	23,812	173,040
	2010	2,299,998	19,457,000	0.138	0.282	21,598	103,582
	2011	0	3,040,086	0.129	0.255	2,962	14,671
Total		**284,216,295**	**218,559,974**	**0.209**	**0.626**	**360,524**	**2,978,911**

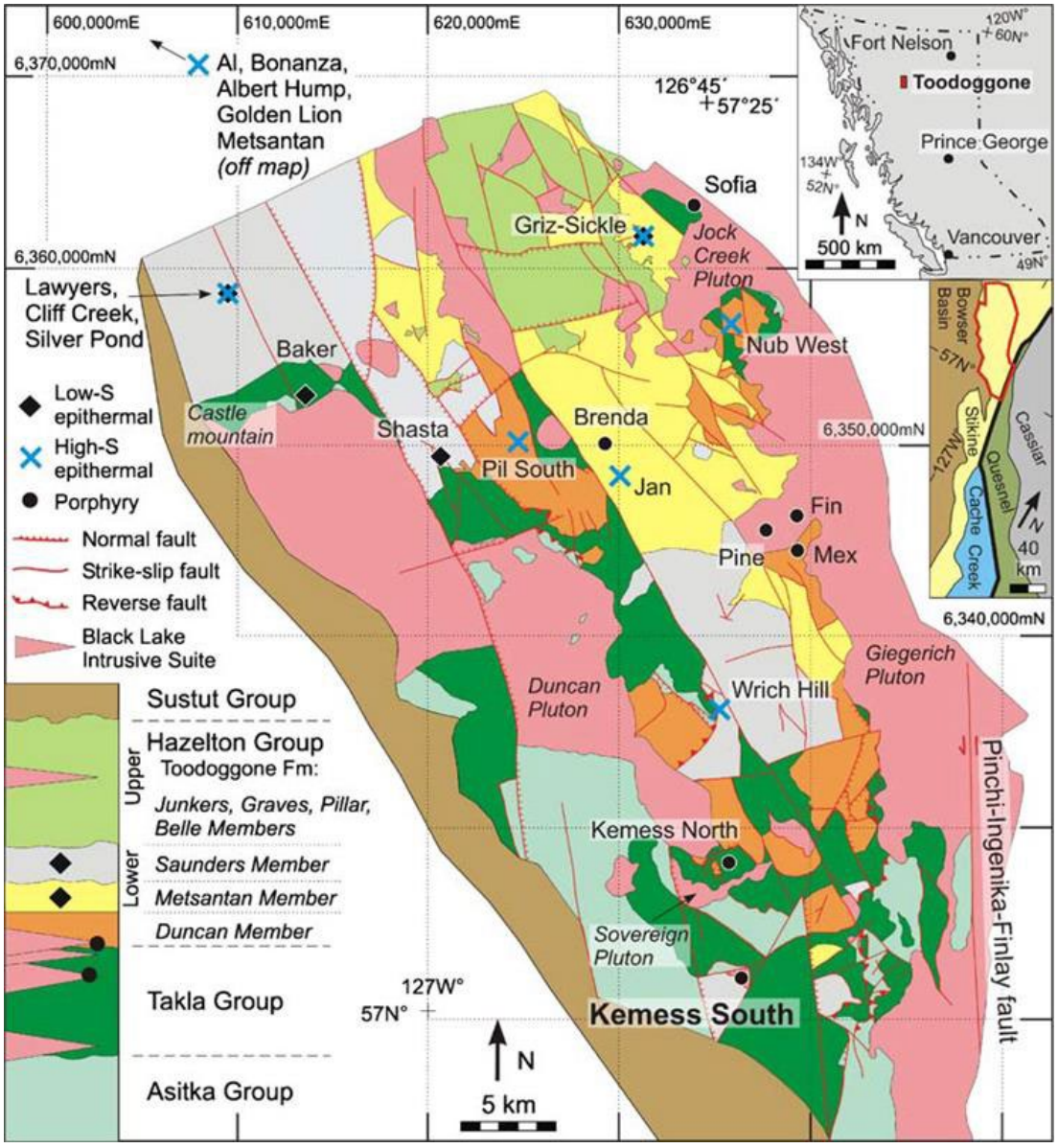
All figures are shown on a 100% production basis without stream agreement deductions.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Kemess project is located at the northeastern margin of the Stikine Terrane of the Intermontane Belt within the prospective Toodoggone district in British Columbia (Figure 7-1). The region is underlain by a northwest-trending 100 km long and 40 km wide belt of Paleozoic and Mesozoic island-arc volcano-sedimentary assemblages intruded by Late Triassic to Jurassic intrusive suites. The volcano–sedimentary rocks in the Toodoggone district are bounded by regional, north-northwest trending, steeply dipping faults that are interpreted as major intra-arc extensional structures that formed within a broader magmatic arc (Diakow et al., 1993).

Figure 7-1: Geological map of Toodoggone mineral district



Geology Map and Stratigraphic Section from Duuring et al., 2009

The Stikine and Quesnel island-arc terranes are host to many significant calc-alkaline and alkaline porphyry gold-copper-silver ± molybdenum deposits and related skarn and epithermal deposits (Figure 7-1). Mineralized porphyry systems in the Toodoggone district are hosted by intermediate-to-felsic plutons of the Black Lake intrusive suite (Late Triassic to Early Jurassic) intruding sedimentary and volcanic rocks. Examples within the region include the Kemess, AuRORA, and Joy. The Toodoggone mineral district is one of the few districts in British Columbia that hosts both significant high-sulphidation and low-sulphidation epithermal-type mineral deposits and occurrences (Diakow et al., 1991, 1993; Duuring et al., 2009). Examples within the region include the Shasta, Baker and Lawyers low-sulphidation epithermal gold-silver deposits. The Ranch project hosts multiple high-sulphidation deposits. Epithermal systems in the Toodoggone are generally younger than the porphyries, although research suggests a possible genetic link between plutonism, porphyry mineralization, volcanism, and high-sulphidation epithermal mineralization (Diakow et al., 1991; Duuring et al., 2009; Bouzari et al., 2019).

The Kemess region is made up of four main lithostratigraphic units including three volcano-sedimentary island arc assemblages referred to as the Asitka Group, the Takla Group, and the Hazelton Group. The island arc assemblages are intruded by felsic to intermediate plutons and dykes of the Black Lake suite (Diakow et al., 1993; Mortensen et al., 1995; Diakow, 2001).

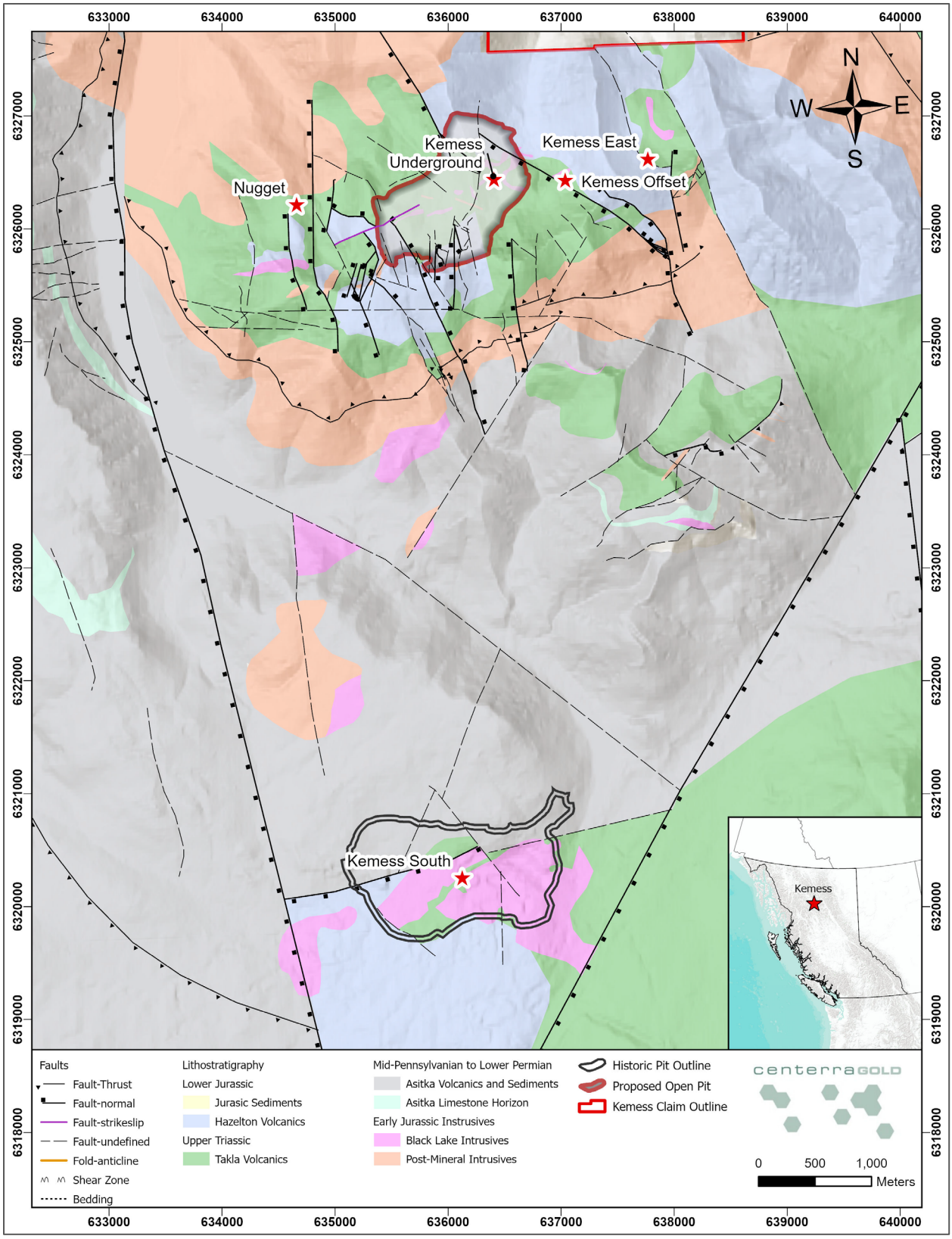
Within the Kemess property (four mine leases and mineral tenure package), there are several known porphyry copper-gold deposits including Kemess South, Nugget, KUG, and Kemess East, as well as several other mineral prospects and showings.

7.2 LOCAL AND PROPERTY GEOLOGY

The Kemess area is made up of four main lithostratigraphic units including three volcano-sedimentary island arc assemblages (the mid-Pennsylvanian to Lower Permian Asitka Group, the Upper Triassic Takla Group, and the Lower Jurassic Hazelton Group) intruded by Upper Triassic to Lower Jurassic felsic to intermediate plutons and cogenetic dikes of the Black Lake suite (Diakow et al., 1993; Mortensen et al., 1995; Diakow, 2001). The Asitka is unconformably overlain by Takla Group rocks, which are in turn unconformably overlain by the Hazelton Group (Diakow et al., 1993) (Figure 7-2).

The Black Lake suite comprises the plutonic rocks observed on the Kemess property, including syn-mineralization and post-mineralization intrusions. These Black Lake suite intrusive rocks are associated with most known mineralized porphyry systems in the Toodoggone district (Duuring et al., 2009). Unmineralized plutons of the Black Lake Intrusive Suite are generally younger, display equigranular textures and are exposed over larger surface areas than the mineralized plutons, although mineralogical and chemical characteristics are comparable.

Figure 7-2: Geological map of the Kemess deposits



7.3 LITHOLOGY

7.3.1 Asitka Group

The Asitka Group forms the basement of Kemess and represents the oldest known rocks on the property (Figure 7-2). Asitka Group rocks exposed on the property consist of two primary units including the Lower Volcanic unit made up of mafic to intermediate volcanic flows and tuffs, and the Upper Sedimentary unit made up of a variety of sedimentary lithologies, including an extensive limestone horizon near the upper contact.

The Lower Volcanic unit is composed primarily of submarine basalts, basalt-derived epiclastic mudstones and lesser mafic to intermediate tuffs. Most of this unit is massive with rare amygdaloidal, pillow, and augite-phyric versions and thin silt to very rare sandstone horizons. The units are overlain and intercalated with sedimentary to tuffaceous rocks ranging from basaltic to rhyolitic in composition. The sedimentary rocks in this middle horizon are dominated by siltstones and with lesser poorly sorted primitive conglomerates. There is at least one significant limestone horizon near the upper contact of Asitka.

Asitka basalts and tuffs are occasionally pyroxene and feldspar-phyric particularly higher in the sequence, as seen on Duncan Ridge, and can appear very similar to the overlying Takla Group rocks. Discontinuous rare chert horizons are seen above the limestone in the stratigraphy near the upper contact with Takla Group rocks in the eastern portion of the Orion target zone but have not been noted elsewhere on the property. Siliceous pyritic mudstones are commonly seen in exposures of the Upper Asitka.

7.3.2 Takla Group

The Takla Group rocks have a small footprint of surface outcrop on the property compared to the surrounding Asitka and Hazelton rocks but host a significant portion of the gold-copper mineralization, including the KUG and Kemess East deposits (Figure 7-2). The Takla Group represents a 1 km thick succession of andesitic flows and volcanic breccias and exhibits textures ranging from fine-grained and massive to porphyritic with medium-grained to very coarse subhedral augite ± feldspar phenocrysts.

At Kemess, the Lower Takla successions are difficult to distinguish from the underlying Asitka rocks and are composed of black mudstones, massive aphanitic basalts and lesser augite-phyric basalts. While the Takla flows tend to be volumetrically more porphyritic and the Asitka basalts tend to be more aphanitic, both Takla and Asitka contain augite- feldspar-phyric and aphanitic flows.

The bulk of the Takla sequence is comprised of massive flows of basaltic to andesitic volcanic rocks, the most distinct of which are the Bladed Feldspar Porphyry and Augite Andesite Porphyry but also

include aphanitic basaltic and andesitic flows which may have amygdules, lesser pillow basalts and rare pyroxene-bearing epiclastic sandstones, which are likely derived from the volcanic rocks.

The Augite Andesite Porphyry is generally green in colour and is characterized by abundant augite phenocrysts (up to ~25%) ranging in size from several millimeters up to 5 mm. The augite-only end member is not as abundant and often grades into feldspar phenocryst-bearing versions.

The Bladed Feldspar Porphyry is characterized by abundant, randomly oriented, large plagioclase laths, up to as long as 2 cm, in a dark green to maroon matrix. This unit was observed almost exclusively near the top of the Takla sequence near the unconformity with the overlying Hazelton Group rocks and as clasts within the Hazelton.

7.3.3 Hazelton Group

The Hazelton rocks in the Kemess area are dominated by volcanic breccias, tuffs, interbedded epiclastic sedimentary rocks and/or pyroclastic flows of the Toodoggone Formation. Many of the Hazelton rocks, particularly where they appear to be fresher, have a distinctly maroon colour which may indicate shallow to subaerial deposition.

The most recognizable Hazelton lithology on the Kemess property is the monomictic to polymict dacitic volcanic breccia to crystal tuff. It is characterized by poorly sorted to unsorted pebble- to boulder-sized clasts (ranging from 1% to 30% of the rock) that are dominantly composed of a feldspar-phyric andesite to dacite. The groundmass in these fragmental rocks is composed of fine-grained feldspar (10–30%), quartz (5%), and magnetite as an accessory. While the breccia can be almost monomictic, other clast populations are most often present, including andesitic hornblende-phyric clasts, maroon mudstone clasts, bladed feldspar porphyry, and lesser pumice fragments. Near the base of the Hazelton, these clasts are often dominated by large Takla fragments including Augite Andesite Porphyry and Bladed Feldspar Porphyry clasts. The clasts are often more strongly altered than the surrounding groundmass giving the unit a mottled appearance. The breccia units have gradational contacts with finer epiclastic and pyroclastic andesitic to dacitic tuffaceous rocks.

The tuffaceous fragmental rocks of the Hazelton units tend to be thick monotonous sequences with lithic fragments of Takla basalt and porphyritic intrusive rocks (similar to Black Lake rocks) suspended in siliceous matrix with rare quartz eyes.

Overall, the Hazelton Group rocks on the property are weakly propylitically altered with hematite staining, weak epidote and zeolite-calcite veining and alteration with weak to moderate silicification. In the areas of known mineralization, such as KUG and Kemess East, the alteration of the Hazelton units intensifies in proximity the faults that separate it from the Black Lake suite and Takla Group. Locally it may have intense quartz-sericite-pyrite phyllic alteration. There is also strong propylitic alteration in

Hazelton Group rocks unconformably above the Takla Group near the Nugget zone. This suggests that at least the very earliest of the Hazelton Group rocks predate the latest mineralizing events.

7.3.4 Intrusive Rocks – Synmineral

The Black Lake intrusive suite comprises most of the plutonic rocks observed on the Property, including syn-mineralization and post-mineralization intrusions. They are thought to be the calc-alkaline intrusive equivalents of the Hazelton Group extrusive rocks.

Most of the larger bodies show evidence of multiple intrusive phases including some incidences of intrusive breccia. Geochronology indicates emplacement dates, of the mineralized Black Lake bodies at Kemess South ca. 201 Ma (201.1 ± 1.2 Ma, Kemess South Re-Os age from Duuring et al., 2009a), with similar age dates recorded for KUG (~202 Ma). The mined-out Maple Leaf Pluton at Kemess South is the only known significant mineralized intrusive body exposed at surface on the Property. Mineralized intrusive outcrop on the Property is sparse outside of Kemess South. A few small pyrite-bearing, quartz-pyrite veined dykes with slightly elevated copper content are observed in the KUG and Kemess East zones but no other bodies of significant economic value have been observed on surface within the Property bounds.

Most of the Black Lake intrusive suite phases and dykes observed during surface mapping have a fine-grained groundmass with abundant-to-crowded medium- to coarse-grained feldspar phenocrysts making up more than 50% of the rock. They are variably hornblende, magnetite and biotite-bearing, have generally low quartz content (<15%), and range from monzodioritic to lesser monzonitic compositions (Diakow et al., 2001). Many of the intrusive rocks on the property have distinct pink colouration; generally due to strong hematite staining of the feldspars and not a reflection of high K-feldspar content.

In a regional scale, the Black Lake plutonic suite and several smaller bodies are either elongated or preferentially oriented to the northwest suggesting that the ascent of magma to subvolcanic levels was probably facilitated by extensional structures of similar orientation (Diakow et al., 1993). At Kemess, the structural control on plutonic bodies distribution has been noted, in particular the dykes appear to align with east-west trending extensional faulting that cut all units across the property.

7.3.5 Intrusive Rocks – Post-Mineral

Most of the intrusive rock exposures on surface are post-mineralization dykes. These have significant textural variation, as well as apparent compositional variation, even over short distances. A few of the larger intrusive bodies have groundmass that is fine to medium-grained; most are porphyritic with only rare occurrences of equigranular texture. Two of the most extensively outcropping intrusive bodies on the property are the post-mineral Sovereign Pluton dated at 202.7 +1.9/-1.6 Ma, which outcrops in the

southern slope of the headwall ridge south of Kemess East and KUG, and the Duncan Lake Pluton dated at 197.3 +1.7/-0.9 Ma, which outcrops to the north and west of KUG. Both these plutons are coarse-grained and predominantly equigranular and are almost identical in appearance and composition despite a difference in age dating (Diakow et al., 2001). Both are associated with inferred major fault structures, interpreted as both thrust and normal type. A third post-mineral pluton, the Hilda Pluton, has been mapped in the Hilda target area (Figure 7-2). Felsic dykes cut their respective plutonic rocks and Duncan Member Toodoggone Formation rocks at the Kemess North (McKinley, 2006).

There are two suites of mafic, gabbroic dykes that crosscut other rocks on the property. Both are generally aphanitic to hornblende-phyric. The earlier suite is only observed crosscutting Asitka Group rocks in the South Dam zone. In general, these dykes are similarly altered and cut by the same structures as the surrounding Asitka Group lithologies. Porphyritic varieties often display strong foliation. It is unclear whether these dykes are related to Takla Group volcanism or another event. The later suite, while compositionally identical to the earlier suite, cut across Hazelton and Takla Group rocks. These dykes tend to be small in scale, irregular, subvertical, and cut across or exploit structures. Late northeast-striking gabbro dykes intruded along northeast-striking fractures in Toodoggone Formation units during northeast–southwest directed shortening (Duuring et al., 2009).

7.4 STRUCTURE

The units of the Toodoggone district are bounded and cut by northwest to north-northwest trending kilometre-scale steeply dipping normal faults (e.g. Saunders Fault), some of which also have lesser strike-slip components. Locally, steeply dipping, northeast trending faults appear to truncate and displace these northwest-trending faults; and form steep valleys (Diakow et al., 1993). Both sets of faults are thought to represent major intra-arc extensional structures and are the primary contributors to the characteristic horst-graben fault block structure of the region which juxtaposes panels of varying depth and age over relatively short distances. Smaller scale east-west trending faults, both predating and postdating these larger faults, appear to have a strong control on emplacement of plutons and dykes and are therefore likely to have strongly influenced the localization of porphyry and epithermal mineralization within the region (Duuring et al., 2009).

Less predominant are northwest to northeast trending thrust faults, which locally place older Asitka Group rocks on top of younger Hazelton Group. These thrust faults are thought to be the result of east-west shortening during either the mid-Jurassic or Early Cretaceous associated with the development of the Skeena fold and thrust belt (Nelson & Kyba, 2014). Many of these were likely later reactivated during extensional relaxation events. The degree of folding to faulting is variable across the terrane; the strata of the Toodoggone Formation are not strongly deformed, whereas older Asitka Group rocks are locally strongly deformed. Folding in the Asitka Group rocks and Takla rocks is likely associated with the development of the regional thrust faulting described above (Diakow et al., 1993).

Mineralized intrusions in the Toodoggone can be dextrally displaced along regional-scale northwest-southeast oriented transcurrent faults (Diakow et al., 1993). One example is the Saunders-Wrich Fault to the east of Kemess South and Kemess North (Figure 7-3). These faults are steeply dipping normal faults and are commonly truncated and displaced by high-angle northeast-trending faults which results in variably titled and rotated blocks.

Major faults in the northern part of the Kemess property include the Nugget, Basal and Kemess North Thrust faults. At Kemess South, the major structure that largely controlled the northern pit wall of the mine is referred to as the North Block Fault. The relative sense of motion on this fault is normal and sinistral with a moderately steep dip to the south and is striking to the east, though the absolute amount of offset is unknown (Rogers and Houle, 1999). The most prominent structure traversing the Kemess Underground area is the Kemess North Thrust fault, an east-west trending south dipping reverse fault that truncates the Kemess North pluton and associated mineralization at depth.

At Kemess South, younger northwest and northeast striking normal–dextral faults cut all rock types, orebodies, and the North Block fault, with displacements of <100 m, and result in the graben-and-horst-style block faulting of the stratigraphy and ore body (Duuring et al., 2009) (Figure 7-4). The same style of graben-and-horst-style block faulting of the stratigraphy and ore body is observed along the KUG and Kemess East mineralization trend.

Figure 7-3: Geological map of the Kemess North deposits

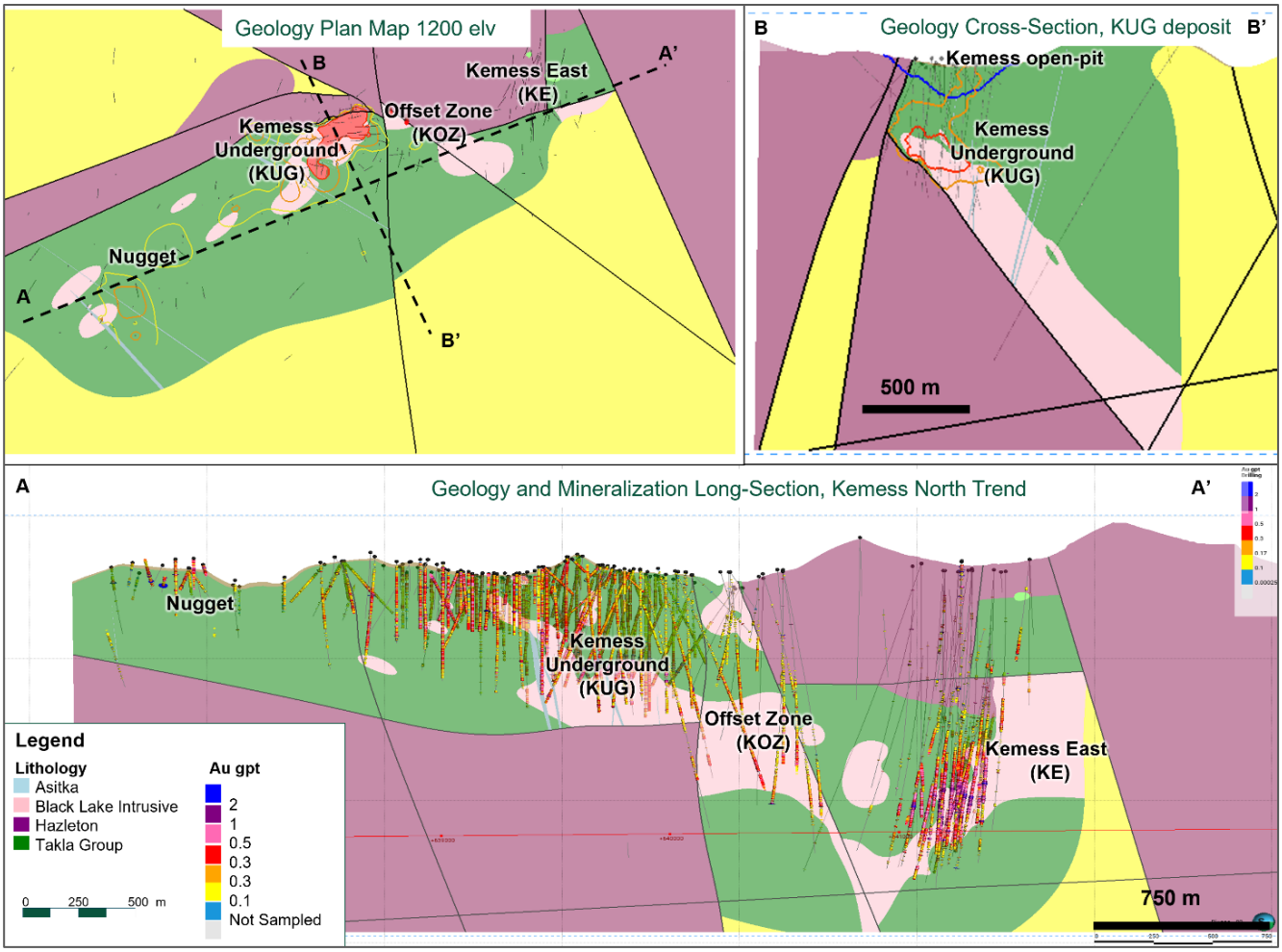
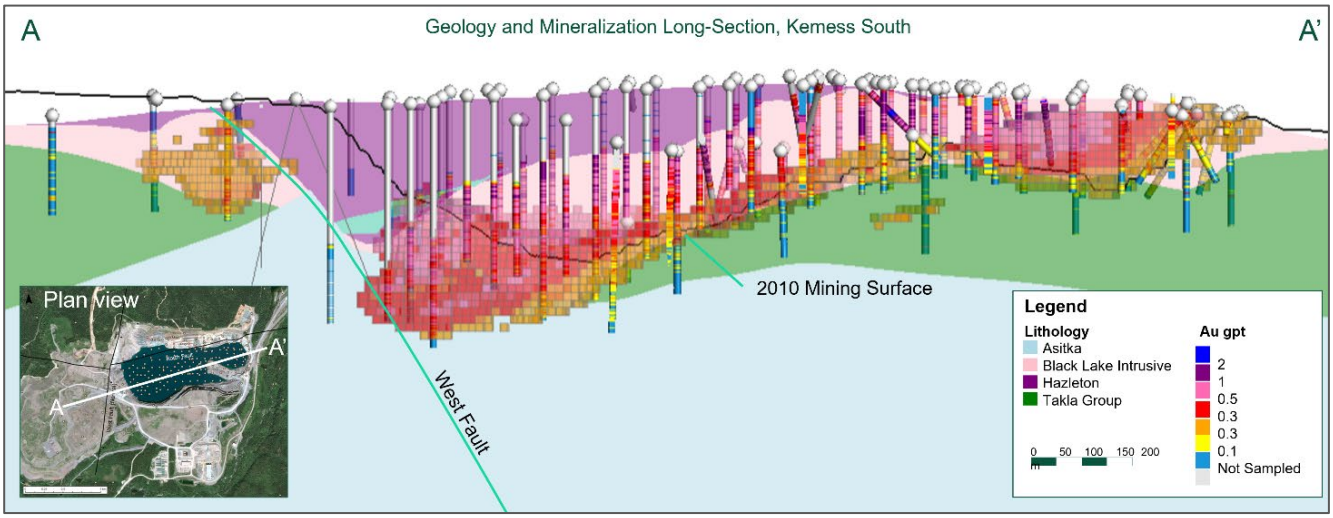


Figure 7-4: Geological map of the Kemess South deposits



7.5 ALTERATION

The major alteration assemblages at Kemess are typical of calc-alkaline porphyry deposits and include potassic, phyllic, propylitic, and argillic alteration zones. The highest copper and gold grades are encountered in the potassic+sulphide and potassic+sulphate domains with next highest grades in the potassic+magnetite+sulphide domain. In addition to the potassic alteration zones, phyllic alteration domains host both copper and gold mineralization.

Geochemical analysis of multi-element drill core data was completed for the KUG deposit by CSA Global in 2020, and for the Kemess East deposit by electromagnetics in 2024, who purchased CSA Global. At KUG, multiple phases of hydrothermal alteration were identified. The dominant alterations styles are sericite-phengite (phyllic assemblage), chlorite-illite, and quartz-magnetite (potassic assemblage). Much of the logged potassic alteration appears to have been retrogressed to phyllic facies in subsequent alteration events.

Analysis of short-wave infrared (SWIR) data from drill core in the KUG and surrounding area highlighted relationships between copper-gold mineralization and the spectra of various alteration minerals. The results support the concept that SWIR data may be used to differentiate hydrothermal environments of the Kemess porphyry system and particular alteration minerals and mineral assemblages (i.e. Illite, illite-chlorite, illite-kaolinite, illite-dickite) have a stronger spatial relationship with copper-gold metal endowment and may be used as vectors in future exploration programs.

SWIR data at KUG shows common occurrences of early potassic alteration overprinted by later phyllic or chlorite-sericite alteration. Typically, the former is only preserved as small (5 cm) pods of preserved magnetite+biotite+potassium feldspar in a domain where quartz+sericite+pyrite±chlorite is dominant.

Common with the other Kemess deposits, several phases of hydrothermal alteration were also identified geochemically at Kemess East. A deep potassic alteration zone, mostly restricted to the quartz monzonite units, is surrounded by phyllic alteration zone (characterized by sericite+pyrite). These mineralization alteration zones occur below propylitic and chlorite-carbonate alteration at shallower levels. Retrograde overprint of potassic alteration by phyllic alteration may have changed original geochemical signatures. Magnetite is rare in the phyllic alteration zone, suggesting magnetite destruction with iron being re-precipitated as pyrite.

7.5.1 Potassic Alteration

The diagnostic minerals associated with potassic alteration include biotite, magnetite, quartz, and potassium feldspar. Mafic minerals are replaced by biotite and magnetite and feldspars are recrystallized such that they no longer have cleavage but remain hard. It is notoriously difficult to distinguish secondary biotite, especially at Kemess where it is typically black to dark green and easily

mistaken for chlorite. The strongest potassic alteration corresponds to significant added iron and silica to form a rock that becomes dominantly magnetite and quartz.

In the eastern part of the system, the potassic zone is primarily restricted to the mineralized sections of the Kemess East stock, but weak potassic alteration of Takla volcanics is observed in the eastern portion of the deposit. The potassic alteration is dominated by secondary biotite, chlorite, quartz and minor sericite. Higher grades of copper-gold mineralization are associated with strong biotite and secondary quartz alteration. Potassic alteration at the higher-grade underground portion of the KUG deposit includes overprinting of weak sericitization (phyllic) alteration, with weak to strong sericitization of potassic alteration at Kemess East.

7.5.2 Chlorite-Sericite Alteration

Chlorite-sericite alteration is similar to phyllic alteration but with the distinct addition of forest green chlorite alteration of mafic minerals. Where pervasive, this alteration assemblage is also magnetite destructive. Chlorite-sericite alteration is common at the Kemess Main area, both at depth surrounding potassic alteration and at shallower levels peripheral to phyllic zones.

7.5.3 Propylitic Alteration

Propylitic alteration occurs in the distal parts of the porphyry system. The assemblage is characterized by epidote and chlorite with pyrite, magnetite, and/or specular hematite. It is generally not texture destructive and associated with late veins, including carbonate veins. The propylitic zone extends from the Takla Group and into the Hazelton Group.

7.5.4 Argillic Alteration

Argillic alteration is dominated by clay minerals such as kaolinite, halloysite, smectite, and quartz. Argillic alteration is produced by highly acidic fluids that can form from the most evolved fluids associated with a porphyry hydrothermal system (hypogene argillic alteration) or from weathering of sulfides at surface (supergene argillic alteration). Both are present at Kemess. There is one diagnostic texture related to hypogene argillic alteration that has been recognized at the KUG deposit, Gusano, consisting of irregular ‘worms’ of pyrite and pyrophyllite up to about 3 cm wide in a dominantly silica matrix. Advanced argillic alteration with abundant clay minerals typically forms in epithermal environments above the porphyry deposits (Sillitoe, 2010). The argillic alteration is well-preserved at the top of Kemess East deposit including horizons of kaolinite and illite. Advanced argillic alteration occurs in an east-west belt partially overlapping the northern limit of the KUG deposit. This domain includes good examples of gusano texture and dumortierite, both diagnostic of hypogene advanced argillic alteration.

7.5.5 Phyllic Alteration

Phyllic alteration is very widespread at KUG, especially at shallower levels, but much less developed at Nugget. It consists of quartz, sericite, and pyrite. Depending on intensity it can be texture destructive but may also include selective sericite+quartz replacement of felsic minerals and pyrite replacement of mafic minerals including magnetite. There is a broad variation between very white sericite close to fluid upflow zones to pale green sericite in peripheral zones which can be a useful tool for exploration vectoring.

At Kemess East, the phyllic zone is vaguely defined and may extend over thick intervals. Alteration intensity varies from moderate to strong with increasing chlorite and sericite alteration with up to 3–5% pyrite. The Black Lake intrusions show intense phyllic alteration with sericite, chlorite and quartz and to 10–15% pyrite, in places completely obliterating the original lithology. At shallower levels of the KUG deposit the sericite-chlorite-anhydrite alteration shell is associated with low grade (>0.25 g/t Au) mineralization.

7.5.6 Sulphate Leach – Broken Zone

A near surface flat-lying zone of intensely broken rock and rubble, historically referred to as the broken zone, occurs above the KUG deposit. This zone is better described as the sulphate leach zone and is a result of hydration of anhydrite to gypsum followed by dissolution of the sulphate and carbonate minerals by acidic ground waters. The hydration of anhydrite to gypsum corresponds to a 60% volume expansion, resulting in fracturing of the rock mass. Surface weathering of pyrite generates acidic groundwater which leaches the sulfates, carbonates, and causes variable clay alteration. Pyrite is still present at surface in this zone. The zone averages a thickness of about 80 m from the surface to competent bedrock and corresponds to angular rubble with variable clay alteration overprint on hypogene assemblages.

The sulphate leach unit is defined in multi-element geochemistry data by a distinct ICP detectable calcium depletion which is directly related to the complete absence of any sulphate minerals (anhydrite and gypsum) above this interface. The interface between the sulphate leach zone and the underlying competent bedrock generally sharp. The sulphate leach zone is the product of present-day weathering processes. The post-mineral porphyritic feldspar dykes remain unaltered and competent within the leach zone due to the lack of anhydrite veins crosscutting the dykes.

7.6 MINERALIZATION

Within the Kemess property, there are several known porphyry copper-gold deposits including Kemess South, Main Zone (which includes Nugget), KUG, and Kemess East. The Kemess Main Zone deposit lies approximately 6 km north of the existing Kemess South processing plant and other mine infrastructure (Figure 7-2). The Main Zone deposit is a near surface porphyry deposit with low-grade ore

zone starting near surface on its western side, and a higher-grade zone between 300 m and 550 m below surface on the eastern side. The deeper, higher-grade ore is the focus of the KUG mine project.

The KUG deposit is in the central portion of the Kemess North ridge porphyry mineralization trend, with the Kemess East deposit ~1 km east and the Nugget Zone ~1.5 km west. The KOZ is in the adjacent fault block east of the KUG deposit and could be the fault offset of the KUG deposit. The linear porphyry mineralization trend continues to the east, connecting the KUG deposit to KOZ and then to the Kemess East deposit. The Kemess East deposit is deeply buried, and mineralization starts at an average depth of 900 m below surface and extends to 1,500 m below surface. Unlike with KUG, there is no known significant low-grade mineralization associated with Kemess East.

The Main Zone and Kemess East calc-alkaline porphyry deposits host copper-gold-silver and molybdenum mineralization. The highest copper and gold grades occur in the monzonite Black Lake suite intrusions, with some elevated gold and copper in the surrounding Takla volcanic units. The Main Zone deposit is centred on a polyphase mineralized porphyritic quartz-monzodiorite to diorite and hydrothermal breccia complex. The intrusive cluster that hosts mineralization is tilted to the north-northwest. The deposit is unconformably overlain by Hazelton rocks.

At the Main Zone and KUG, the main sulphides hosting gold and copper mineralization are chalcopyrite and pyrite. Other accessory mineral phases include pyrrhotite and a low iron sulphate zone with gypsum/anhydrite and pyrite. The more distal, lower-temperature gold mineralization hosted by Takla rocks in the western part of the project area (i.e. Nugget zone) is generally associated with weaker alteration of the host rocks. Concentrations of lead and zinc are generally very low, so effect of zinc and lead sulphides can be neglected as well as trace amounts of bornite and other copper sulphides.

At Kemess East, higher-grade copper-gold mineralization is characterized by strong secondary biotite and quartz alteration and lesser chlorite alteration in the plutonic rocks. Higher grade copper-gold mineralization is hosted in intrusions and characterized by a slight increase in quartz veining and an increase in chalcopyrite to pyrite ratios. Quartz veining with mineralization is present in the Kemess East potassic zone, but it is not as dominant as at KUG which has intense quartz stockwork veining with mineralization. The pyrite to chalcopyrite ratio within Kemess East is roughly 1:1 whereas in KUG it is 3:1 with the bulk of the copper-gold mineralization hosted within the overlying Takla Group. There is also less magnetite mineralization/alteration within the Kemess East deposit compared with the KUG deposit.

Similar to KUG, the mineralization in the KOZ is hosted within the Takla Volcanics and the Black Lake Intrusive. Mineralization is disseminated and within stringers and veins with mostly pyrite, chalcopyrite and molybdenite. Gold is associated with the chalcopyrite.

7.6.1 Vein Paragenesis

The vein classification scheme for Kemess Main Zone developed by McKinley (2006) was used as the foundation of the vein logging scheme for AuRico-Centerra projects. From McKinley (2006), a more detailed vein logging classification was developed, distinguishing the veins by major and minor vein fill and alteration halo mineral assemblages and following the same groups of early, transitional, and late veins typical of porphyry deposits.

Multiple phases of copper and gold mineralization have been documented at the Kemess South deposit. The first mineralization event is associated with early-stage and vein types with enveloping potassic alteration zones. The secondary and main copper-gold-molybdenum mineralization event is associated with stage 3 veins (quartz-pyrite-chalcopyrite-magnetite, now called transitional type) and coincides with the transition from potassic to phyllic or intermediate argillic alteration (Duuring et al., 2009). Early potassic alteration and early-stage copper ± gold ± molybdenum mineralization is commonly replaced by phyllic and intermediate argillic alteration related to the main-stage copper-gold-molybdenum mineralization event. Late-stage pyrite-rich stringer veins are cut by post-mineralization stage anhydrite-rich, carbonate-rich, and chlorite veins (Duuring et al., 2009).

At the KUG deposit area, early veins are dominated by magnetite, quartz, and chlorite. Some early veins have albite halos, indicative of high temperature. Most of the pyrite and chalcopyrite mineralization is associated with transitional vein types including subtypes of quartz+pyrite+magnetite+chalcopyrite veins, and pyrite dominant veins with sericite halos. These veins are typically 1–10 mm wide with an irregular/discontinuous form and halos can extend 5–10 times the width of the vein. Weak copper-gold mineralization is also associated with late-stage vein types including quartz-calcite-pyrite and calcite-magnetite veins which may have potassic halos.

Plots of gold and copper vs vein types at KUG show that economic concentrations of metals (values >0.5 g/t Au and >0.3% Cu) are spatially related to groups of veins that include transition veins and/or late veins. In addition, high-grade sample intervals are associated with complex vein groups that include multiple generations of veins (i.e. assemblages of early, transitional and late veins present in a single interval), potentially indicating a preferred hydrothermal pathway for fluids during the evolution of the hydrothermal system.

7.6.2 Supergene Mineralization

The sulphate leach zone at surface above the KUG deposit, is effectively a poorly developed supergene zone. Pyrite is preserved to surface and is common throughout (5–7%) as both disseminated and within veins. The sulphate leach zone contains low grade gold and very low-grade copper mineralization and will often show a slight increase in gold with depth. A very minor copper sulphide enrichment occurs near the base of this zone in its western half, corresponding to isolated occurrences of chalcocite. At

Kemess South, there is a well-developed supergene zone. This supergene zone formed during the Jurassic and was only preserved because it was buried beneath younger Toodoggone Formation stratigraphy. The upper portion of the deposit is fully oxidized and contains gold-only mineralization with no sulphur or copper. A narrow secondary enrichment blanket occurs where disseminated sulphides are observed to be partially rimmed by chalcocite, which accounts for the elevated copper grades in the lower portion of the supergene alteration zone (Duuring et al., 2009). Native copper occurs near the base of the oxidized zone.

7.6.3 Chalcopyrite

Chalcopyrite hosts copper and gold mineralization at the Kemess deposits. The mode of occurrence of chalcopyrite is as both veins and disseminations. Disseminations of chalcopyrite occur in higher density in zones of higher density quartz-magnetite stockwork and quartz-magnetite veins. Gold and copper grades variably diminish outward into the hangingwall and footwall volcanics of the mineralized and variably potassic altered Black Lake suite intrusions.

7.6.4 Pyrite

Overall, sulphide mineralization throughout the deposit consists of 2–3% pyrite, with lesser amounts of chalcopyrite and traces of molybdenite. Pyrite is associated with copper-gold mineralization and occurs as disseminations, fracture fillings, and veins up to a few centimetres wide generally associated with quartz-gypsum-magnetite veins and zones of quartz-magnetite replacement. The total sulphide content in the core of the KUG deposit averages 3–5%, rising to 10–12% in the phyllic halo.

At the Kemess East deposit, pyrite within the phyllic zone ranges from 3% to 5% and is disseminated and veined. Minor quartz veining is present within the phyllic zone with pyrite ± chalcopyrite. This is different to KUG where there is a sulphate leach zone that forms a broken zone near surface and lower phyllic zone high in gypsum veining. The phyllic zone in KUG contains significantly more pyrite mineralization than Kemess East.

7.6.5 Molybdenum

At both Main Zone and Kemess East, the potassic zone contains a high percentage of the copper-gold mineralization with an upper zone of molybdenum mineralization. Molybdenite is present in the transition zone from phyllic to potassic alteration and is present with chalcopyrite and pyrite within quartz veins, late-stage zeolite and carbonate veins, and within joints.

7.6.6 Length Width Continuity

The Kemess deposits are roughly tabular, titled, calc-alkalic gold-copper porphyry deposits that measure approximately 650 m north-south, 1,500 m east-west, and extend to a vertical depth greater than 1,500 m. The three principal deposits, Kemess South, Main Zone, and Kemess East, are centred

on mineralized composite monzodiorite or granodiorite porphyry stocks. Within this system, the overall shapes of the mineralized bodies are variable and offset along major structures.

The Kemess North mineralization trend is an east-westerly striking prospective belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east, the trend includes the Nugget, KUG, Kemess Offset, and Kemess East deposits, with a general trend of deposits occurring at greater depths towards the east.

The Main Zone area, extending over 2.5 km east-west, and 600 m north-south, consists of a low-grade ore zone at a depth of 0–300 m below the surface on its western flank (including the Nugget zone and Kemess open pit) and a higher grade zone (values >0.5% Cu and >0.5% g/t Au) from 300–550 m below surface on the eastern side, which forms the KUG project. The low-grade western portion of the deposit is hosted by altered Takla Group volcanics with Black Lake intrusive suite dykes (10–100 m wide), that extend into the Nugget zone. The highest grade of the KUG deposit is centered on a mineralized Black Lake porphyritic monzodiorite/diorite pluton that dips shallow to moderately south.

The Kemess East deposit, approximately 1 km east of KUG, is deeply buried and mineralization starts at an average depth of 900 m below surface and extends to 1,500 m below surface. Unlike KUG, there is no known significant shallow low-grade mineralization associated with Kemess East.

The known deposit extents, particularly at Kemess East, are limited by the lack of drill hole information and remain open to the east along the linear porphyry trend of the stock cluster.

8 DEPOSIT TYPES

The Kemess deposits are categorized as calc-alkalic copper-gold porphyry deposits and are recognized in only a few mineral provinces worldwide (Deyell and Tosdal, 2004). Within the Canadian Cordillera, calc-alkalic deposits are more than twice as common as alkalic ones, with more than 1000 occurrences in British Columbia and the Yukon. Deposits formed during the brief Triassic-Jurassic mineralizing epoch in Quesnellia and Stikinia accounted for ~75% of the metallic mineral production in British Columbia in 2008 ($2.516 billion total) and 100% of Yukon metal mining production in 2009, excluding placer (Logan and Mihalynuk, 2014).

Porphyry and epithermal deposits in the Intermontane belt in British Columbia are genetically linked to volcanic arc terranes that accreted to the margin of ancestral North America. Two arc terranes account for the bulk of the accreted crust and associated porphyry mineralization, the Quesnellia and the Stikinia terrane, host to the Toodooggone mineral district (Logan, 2013).

Porphyry copper and epithermal gold-silver deposits typically form in the upper parts of large magmatic-hydrothermal systems that result from fluids generated from the crystallization of intermediate composition calc-alkalic igneous magmas in island and continental margin arcs (Sillitoe, 2010). A favourable environment for the generation of porphyry copper and related epithermal deposits in the Toodoggone existed throughout much of Late Triassic – Early Jurassic due to a combination of favourable traits including thick volcanic host rock sequences, coeval felsic intrusions and an active structural environment (Bouzari et al., 2019).

Porphyry copper ± gold deposits commonly consist of vein stockworks, vein sets, veinlets, and disseminations of pyrite, chalcopyrite ± molybenite, bornite that occur in large zones of economic bulk-mineable mineralization within porphyritic igneous intrusions, their contact margins, and adjoining host rocks. The mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks (Sillitoe, 2010). Mineralization in calc-alkalic porphyry systems is dominantly hosted in veins or breccias and rarely occurs as disseminations (Figure 9-4). The bulk of mineralization at Kemess is vein-hosted, which is typical for porphyry copper deposits, particularly calc-alkalic types, that are stockwork-hosted deposits with a long-lived history of veining events, typically classified in various generations from early-stage to late-stage (e.g. Sillitoe, 2010).

Porphyry copper deposits have a geochemical footprint of trace metal zonation that is upward and outward from the deeper central copper zone (Halley et al., 2015). Calc-alkaline porphyry systems generally have large footprints, and distal expressions of hydrothermal activity may result in the formation of alteration up to 10 km from the site of porphyry mineralization (Titley et al., 1986; Halley et al., 2015). The alteration and mineralization assemblages typical of calc-alkalic porphyry systems

include potassic, phyllic, propylitic and argillic hydrothermal alteration assemblages, as well as quartz-rich stockworks (Rees and Robertson, 2010).

9 EXPLORATION

Since acquiring the Kemess Project in January 2018, Centerra has conducted various exploration programs in 2019, 2020, and 2024, at both Kemess Main Zone and Kemess South, including geophysical surveys and diamond drilling campaigns. This Item focuses on components of the recent 2019 to 2024 exploration programs other than exploration drilling. Detailed information on exploration drilling programs from 2019 to 2024 is included in Item 10 (Drilling). Information on exploration programs prior to 2019 is covered in Item 6 (History).

9.1 RECENT EXPLORATION ACTIVITIES (2019–2024)

9.1.1 Centerra-AuRico Exploration (2019 to 2024)

Equity Exploration Consultants Ltd worked on the Kemess Project on behalf of Centerra between March 2019 and April 2020. The work was carried out to evaluate the Nugget and KUG deposits at the Kemess North mineralization trend, as well as at a geophysical target in the Kemess South area east of the past producing Kemess South mine.

Prior to carrying out diamond drilling, a relogging campaign of historical drill core was conducted to familiarize the geological team with the Kemess Main and Kemess South deposits, and to develop an updated core logging schema for the complex multiphase hydrothermal systems including alteration and vein classification. This program included re-logging 12,128.5 m of core at Kemess Main including the KUG and Nugget zones, and an additional 3,911.8 m of drill core from Kemess South. The historical drill core relogging was completed in two main phases. Phase I was focused on core from the Kemess Main deposits hosted within the northern area of the Kemess property. This work followed the construction of the relogging work camp from April 1 to June 4, 2019. Phase II was focused on core from the Kemess South area and included holes from both the Kemess South pit area and holes east of the pit. The Kemess South relog program took place from August 14 to September 19, 2019.

The purpose of the relogging at Kemess South was to help refine the drill holes testing the coincident airborne magnetic low, ground IP chargeability highs and conductivity highs apparent in the geophysics data, referred to as the “Eastern Extension” target. The relog program characterized the geology in the eastern margins of the Kemess South pit, determined how the geology changed from west to east from mined out portions of the pit to holes drilled above the eastern chargeability high target (section KS 10150N, mine grid) and provided geological context for the 2019 planned exploration drill holes testing the eastern target.

Exploration activities in 2020 included geophysics ground Induced Polarization ‘IP’ surveys and diamond drilling. Peter E. Walcott & Associates Limited undertook line establishment and induced polarization surveying over portions of the Kemess Mine property between September 14 and 30, 2020. The line

establishment and induced polarization survey consisted of 15 line-km, carried out a total of six north-south traverses including two lines on each of the Rat, Orion and Hilda South greenfield exploration prospects (Figure 9-1).

Figure 9-1: Kemess exploration targets

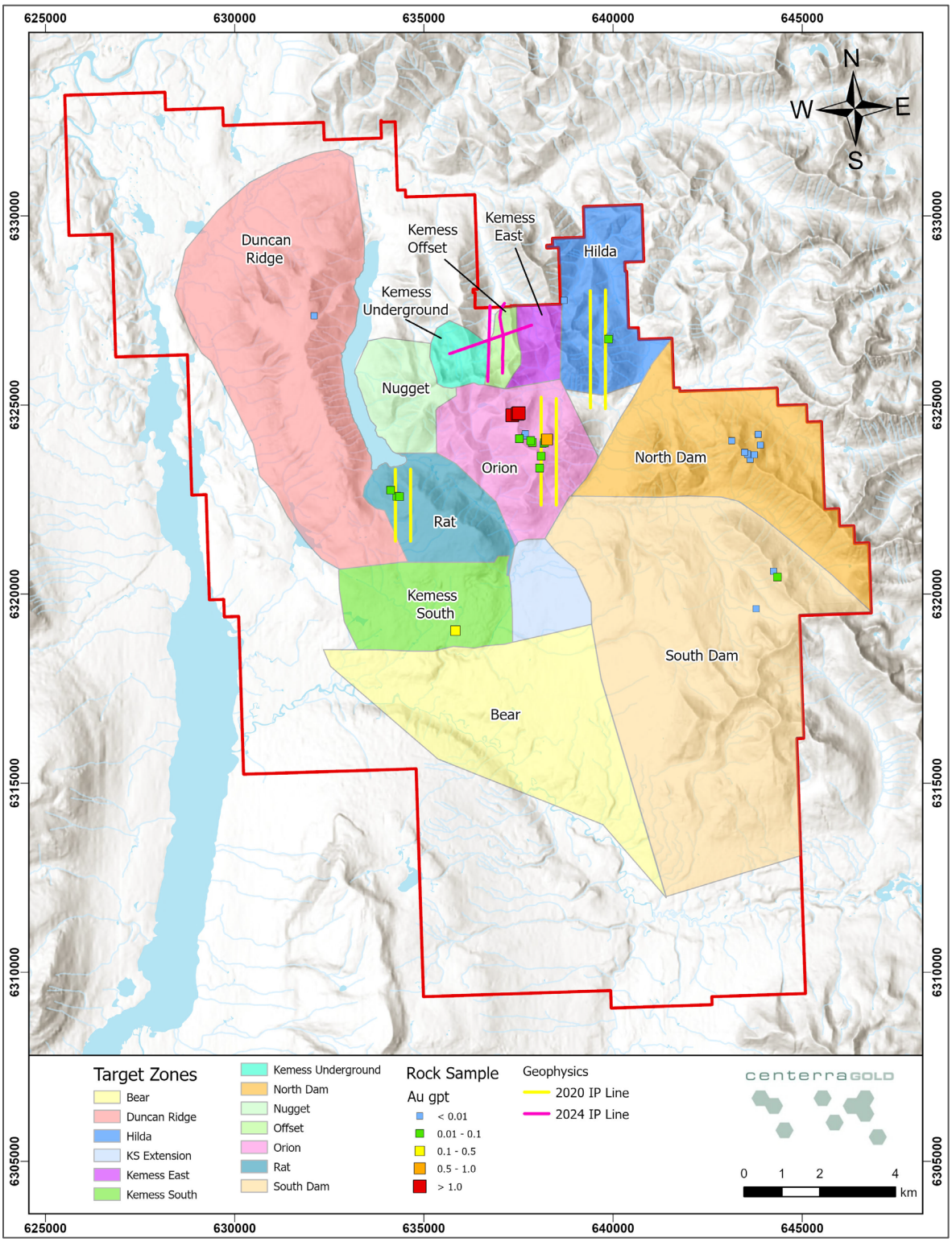
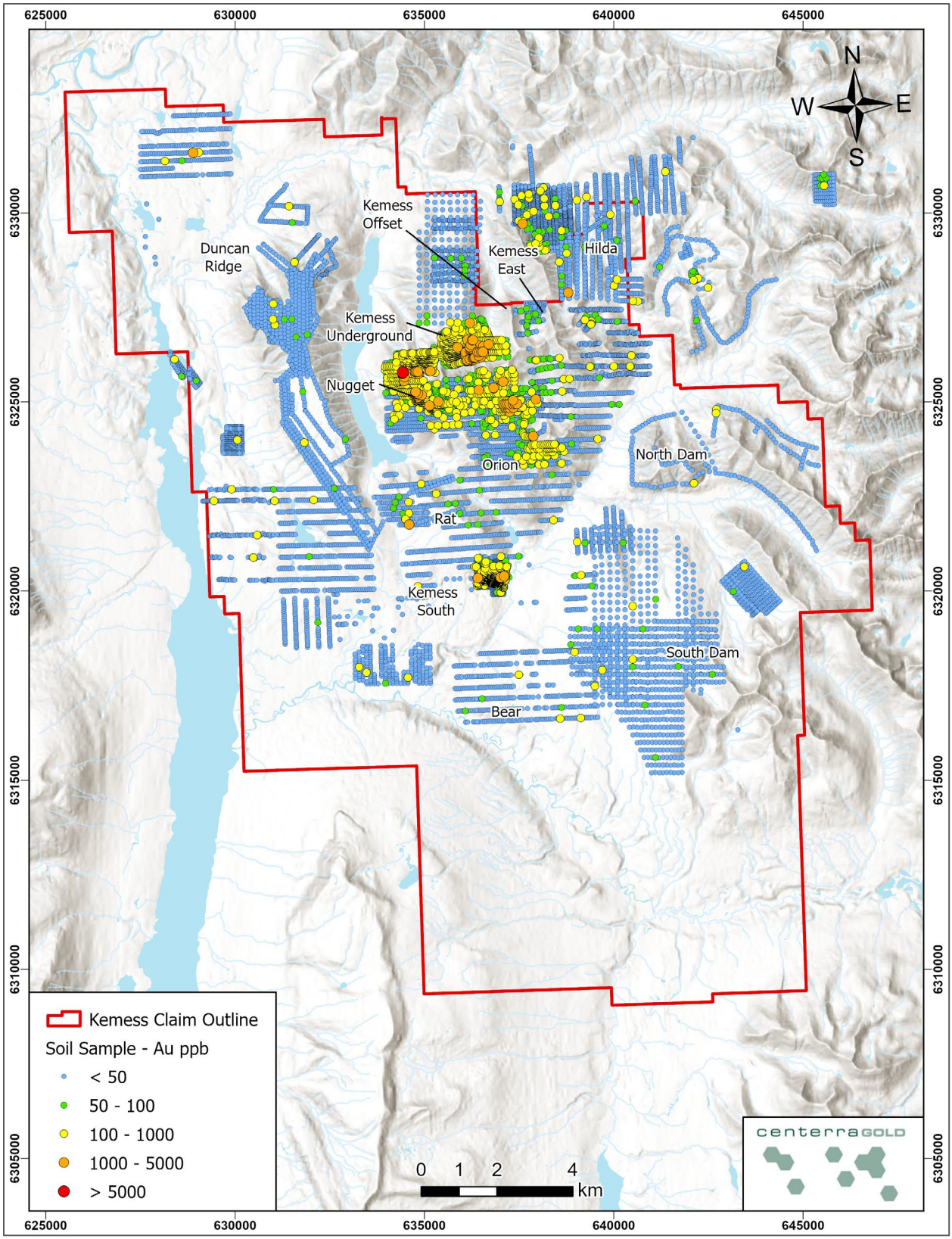


Figure 9-2: Kemess gold soil geochemistry



The survey was carried out using the “pole-dipole” method of surveying with a 100 m dipole employed and 1st to 20th separations readings were obtained. The horizontal position of the stations was recorded using a GPS/GLONASS equipped Garmin C66 handheld global positioning system (GPS) receiver. The data were presented as individual pseudo section plots of apparent resistivity and apparent chargeability generated using Geosoft Oasis Montaj. In addition, data was subjected to 2D inversion.

The 2020 IP surveys on the Kemess property tested for chargeability anomalies in areas with historical gold-copper mineralized surficial material (rock and soil samples), including the Orion zone (Figure 9-1). The results from the 2020 IP inversions show small (50–250 m wide) circular, shallow high chargeability anomalies (30–50 mV) identified at Rat and Orion. These areas are recommended for follow up surficial sampling and infill IP surveying. A large, subtle chargeability shallow anomaly was identified at Hilda South. This indicated continuation to the east of the large, east-west trending chargeability anomaly associated with the Kemess Main Zone trend (Figure 9-1).

In 2024, Centerra re-assessed the historical Kemess data and constructed updated 3D geological and resource models along the Kemess Main Zone trend. This consisted of an updated and integrated lithology, alteration, structural, and resource block models for the various deposits, including the Kemess Main Zone, and Kemess East zones, that were previously modeled separately. The 2024 Leapfrog Geo generated 3D lithological model consisted of five grouped lithologies logged in core including the Takla volcanics, Hazleton volcanics, Blak Lake intrusions, post mineral dykes and post mineral pluton (includes the Sovereign pluton), (Figure 7-3). A 3D updated alteration model was generated based on geological logging, multi-element assay data, and SWIR data consisting of six domains, potassic, potassic with chlorite-sericite overprint, chlorite-sericite, phyllic, leached zone, and weak leached zone. Exploration and infill drilling programs resumed, completing a total of 11,426 m of diamond drilling at the Kemess Main Zone.

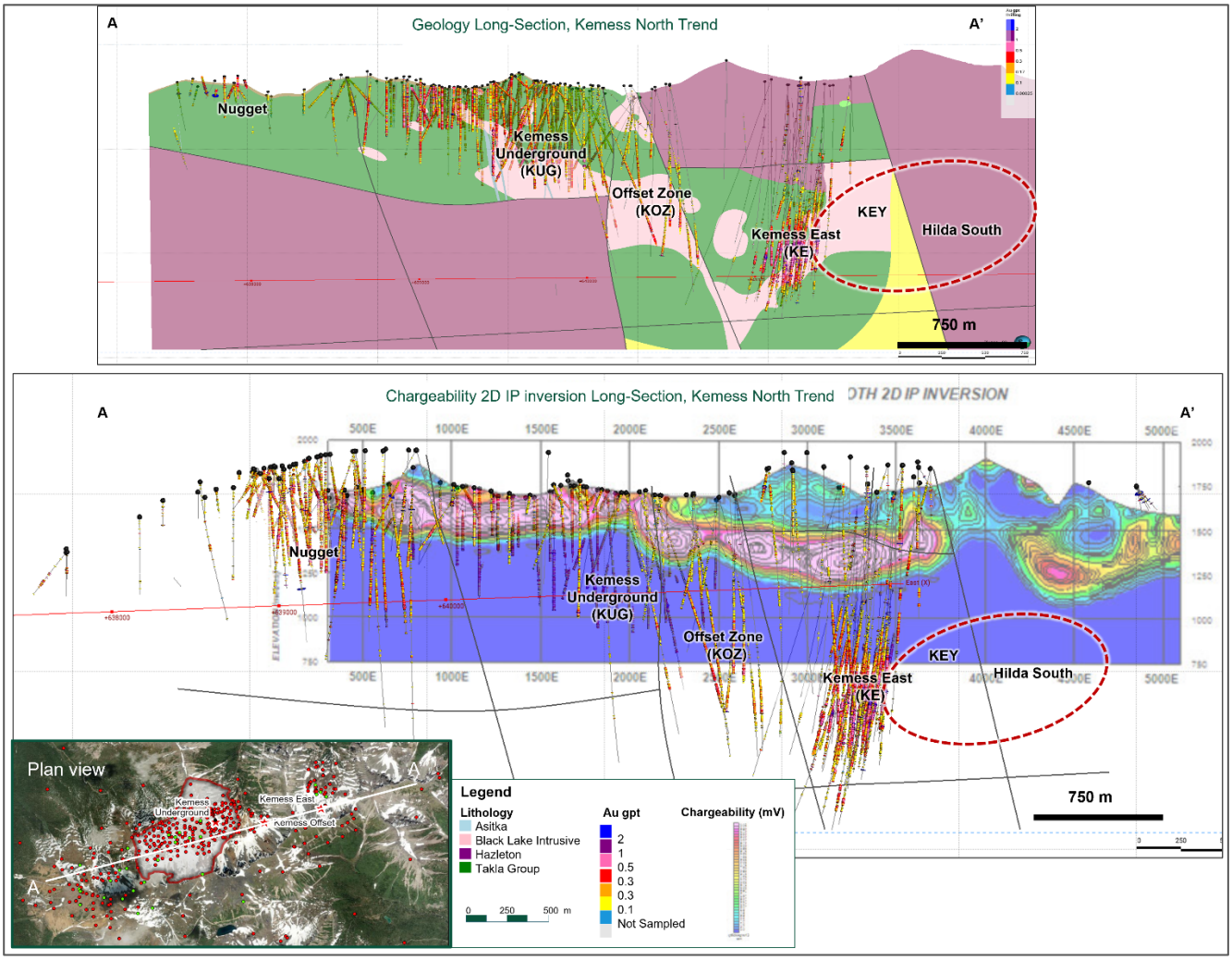
Exploration target development continued along the Kemess Main Zone trend, including a geophysics ground IP survey completed from September 1 to 7, 2024. The line establishment and IP survey consisted of 6.2 line-km, carried out on three lines including two north-south traverses, and one east-west tie-line that extended across the Kemess Main Zone deposit (Figure 9-1). The 2024 survey was completed with the same specifications as described for the 2020 Walcott ground IP survey. The results show a shallow chargeability high anomaly coincident with a resistivity low, with an underlying resistivity high and chargeability low. This is interpreted as a potential pyrite halo in a phyllic alteration zone, overlying a porphyry intrusion at depth, and supports the exploration model for the Offset Zone. The same geophysics signature, a shallow chargeability high overlaying a low chargeability zone that hosts the main mineralization, is observed at the KUG and Kemess East deposits (Figure 9-3).

A description and interpretation of the exploration drill programs completed by Centerra in 2019, 2020, and 2024 are included in Item 10.

9.2 SUMMARY AND INTERPRETATION OF EXPLORATION ACTIVITIES

Recent exploration has continued to define the Kemess Main Zone mineralization trend, an east-westerly striking prospective belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east, the trend includes the Nugget, KUG, Kemess Offset, Kemess East, and Hilda South targets, with a general trend of deposits occurring at greater depths towards the east (Figure 9-3).

Figure 9-3: Kemess North Trend brownfield exploration targets



9.2.1 Nugget Zone – Brownfield Exploration

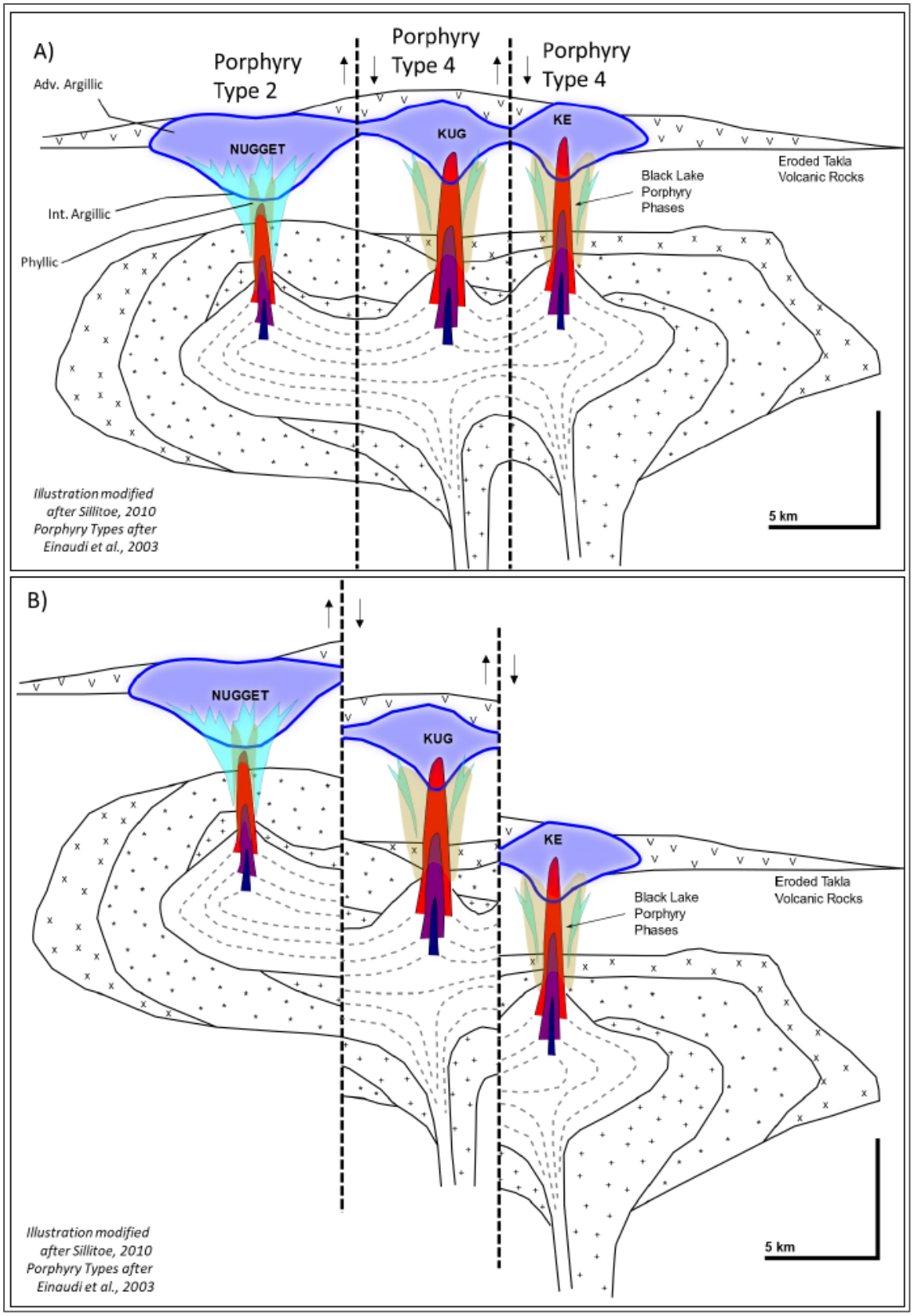
The Nugget Zone is located along the Kemess Main Zone trend, approximately 1,500 m west of the KUG deposit. Nugget represents potentially another mineralized porphyry centre along the Kemess trend, separated from the KUG deposit by a fault (Figure 9-3, Figure 9-4).

Mineralization at Nugget includes relatively narrow zones of high-grade gold at shallow depths (e.g. 8.35 g/t Au and 0.343% Cu over 2 m at 74 m depth in KN-02-56) and broader intersections of lower

grade gold-copper mineralization at depth (e.g. 0.28 g/t Au and 0.13% Cu over 478.65 m starting at 191 m depth in KN-02-49). Most drilling at Nugget has been shallow (<500 m). Alteration assemblages and intersected lithologies (dominantly reported as Takla Group basalt with relatively minor monzonite) suggest that Nugget may represent a higher emplacement position for a porphyry system relative to the KUG deposit.

Exploration work at Nugget in recent years has continued to develop the exploration model at Kemess. An updated genetic model for the porphyry system was interpreted from relogs characterizing the distribution of alteration assemblages and porphyry phases. It is theorized that the difference in characteristics between the KUG and Nugget porphyry systems may be the result of heterogeneities of porphyries sourcing different parts of a common batholith (Figure 9-4, from Equity internal report Lui et al., 2020). Intrusions of the Nugget porphyry are narrow and consist of a higher abundance of breccias in comparison to the KUG porphyry which has more voluminous porphyry stocks. In this interpretation, the Nugget porphyry has more significant advanced to intermediate argillic alteration overlapping with a broad potassic and lesser phyllic alteration distribution. Characteristics of the Nugget and KUG porphyry systems are schematically illustrated in Figure 9-4 where Figure 9-4A illustrates the porphyry systems during emplacement and Figure 9-4B portrays post-emplacement deformation preferentially forming faults between the porphyry centres to define the current structural arrangement of the porphyries.

Figure 9-4: Schematic illustration of the Kemess Main Zone porphyry



Note: Kemess Main Zone porphyry centres during emplacement (A), and post-deformation (B). The illustration of porphyry phases and cooling batholith is modified after Sillitoe (2010). Porphyry “types” follow the generic classification after Einaudi et al., (2005) from Lui (2020).

9.2.2 Offset Zone – Brownfield Exploration

The KOZ is located directly east of the KUG deposit. The KOZ is assumed to be the fault offset of the KUG deposit or could possibly be the extension of the Kemess East deposit. At the eastern edge of the KUG deposit is an interpreted steeply dipping east side down normal fault (Figure 9-3). A similar east side down normal fault is inferred to separate KOZ and the Kemess East deposit, modelled in the 2024 updated lithology and structural model for the Kemess Main Zone trend (Figure 9-3). Majority of the historical drilling at KOZ is interpreted to be too shallow to intersect the bulk of the KOZ mineralization. In 2005, three drill holes intersected the KOZ but the grades were not significant enough to follow up on, due to low metal prices. Deeper drilling was conducted again in 2013 and 2015 in the KOZ following the Quantec Titan Geophysical survey and the ZTEM Airborne survey that show potential of an extension of KUG and/or Kemess East at depth (Figure 9-3). Four drill holes specifically targeted the KOZ in the 2015 drilling campaign, with one hole abandoned near surface due to poor drilling conditions. The best drill result was from KH-15-06A which intersected 817.5 m grading 0.273 g/t Au, 0.216% Cu, 1.43 g/t Ag and 0.006% Mo from 459.0 m depth. This interval included 81.5 m grading 0.437 g/t Au, 0.292% Cu, 1.44 g/t Ag and 0.010% Mo. The KOZ mineralization is hosted in similar host rocks as KUG and Kemess East, Takla volcanics and Black Lake intrusions (Figure 7-3, Figure 9-3).

Exploration work at KOZ continued in 2024 with a ground IP survey that better defined the shallow chargeability anomaly overlying the KOZ, initially identified in the historical Titan geophysical survey, and extended the anomaly to the south. Targeting below this chargeability anomaly, interpreted as a phyllic alteration zone or pyrite halo, can be used to vector towards a potentially mineralized porphyry center and was used to refine locations for planned exploration drill holes at KOZ.

9.2.3 Kemess Main Zone Trend – Brownfield Exploration

Exploration potential exists continuing east of the Kemess East deposit along the Kemess Main Zone mineralization trend, including the KEY and Hilda South targets. These targets have a similar geophysical signature in Induced Polarization “IP” surveys, as the other main deposits including KUG and Kemess East, consisting of shallow tabular chargeability high anomalies, with underlying chargeability lows that host the bulk of the mineralization (Figure 9-3). The anomalies are interpreted to be pyrite-rich phyllic alteration zones (high chargeability-low resistivity) that overly the potassically altered and mineralized rocks. In addition to a similar geophysical signature, similar geochemical zonation occurs over the Hilda South target and the Kemess East deposit; historical shallow drilling at Hilda South intersected arsenic, zinc, and lead anomalies, which are anomalous elements above the main mineralized gold-copper zone at Kemess East. The KEY brownfield target was developed in 2020 from reinterpreting the orientation of the Kemess East Offset Fault that was historically interpreted to cut-off the Kemess East mineralization to the east (Figure 9-3). The KEY target zone may also be

interpreted as the eastern extension of the Kemess East zone. Step out drilling further east from the Kemess East deposit along trend is recommended including the KEY and Hilda South targets.

9.2.4 Orion – Greenfield Exploration

The Orion zone is defined by a significant surface geochemical anomaly, 550 m wide east–west and 350 m wide north–south, with values of 0.15–1.85 g/t Au and 500 to 2,553 ppm Cu to the north and a smaller anomaly of gold up to 0.38 g/t, copper, zinc to the south (350 m wide east–west and 100 m wide north–south) (Figure 9-1, Figure 9-2). These similarities to known porphyry mineralization in the Kemess North trend have generated interest in the Orion prospect. Surface mapping and sampling has identified quartz-carbonate veins hosting chalcopyrite and pyrite within a zone of chlorite carbonate-zeolite alteration (Barnes and Miller, 2018a). Several drilling campaigns, including four 700 m deep holes in 2015 have returned only narrow mineralized intersections within shear zones similar to the quartz carbonate veins at surface (Barnes and Miller, 2018a). The Orion zone is underlain predominantly by sedimentary and volcanic rocks of the older Asitka Group. The chargeability anomaly correlates with graphitic sedimentary rocks intersected in drillholes, which may limit the effectiveness of ground IP surveys in exploration in the target area. The 2024 ground IP over the southern Orion target shows isolated chargeability highs, some associated with low resistivity and some associated with isolated circular resistivity highs. The chargeability highs coincident with the high resistivity features should be followed up as they may be associated with prospective vein hosted targets, rather than the resistivity lows which may indicate argillites.

9.2.5 North Dam – Greenfield Exploration

The North Dam exploration target is north of a major fault that juxtaposes older Asitka to the south with younger Takla/Hazelton volcanics to the north. Soil geochemistry showed a distinct gold ± copper anomaly within a wedge of Hazelton which could indicate the presence of higher level (i.e. epithermal) mineralization (Figure 7-2, Figure 9-2).

9.2.6 Kemess South – East Extension

Located 2 km east of the Kemess South open pit mine is the East Extension exploration zone. This area was drilled in 1991 and 1992 which returned evidence for potassic alteration and copper-gold mineralization. In particular, hole S91-02 is reported as having strong potassic alteration within Takla volcanic rocks and monzonite and returned elevated gold throughout its 191 m depth (up to 0.27 g/t over 2 m at 158 m depth). The geophysics signature of the East Extension zone corresponds to a magnetic low and a chargeability low, similar to the Kemess South deposit. This is in contrast with the Kemess North trend which is defined by a strong chargeability high. The chargeability high is interpreted as the phyllic alteration zone that overlies mineralization along the Kemess North trend. Given that the alteration in the Kemess South East Extension zone is reported as potassic at surface, the lack of an

overlying phyllic zone may explain the lack of a chargeability high associated with the mineralization. This zone has the potential to be an offset extension of the Kemess South deposit and was drill tested in 2019 with no significant gold-copper drill results to report. Results from the 2019 drilling at Kemess South suggest the main chargeability anomaly can be explained by the pyrite-argillite beds as well as the graphitic shear zone intersected from 644.8 to 665.5 m in KS-19-05 whereas the resistivity high can be explained by the siliceous fragmental rocks overlying the graphite and pyrite rich layers, with no significant potassic alteration zone intersected. The source of the chargeability anomaly is likely the disseminated pyrite encountered at the top of the hole (6–8% and locally up to 20%), the sulphide occurrence is interpreted as a primary feature of the lithology and not indicative of gold-copper mineralization.

Little is known about the Hazelton volcanic stratigraphy at Kemess North and Kemess South, which may have exploration implications, for example can be used as marker horizons to construct offsets between fault blocks, and as potential hosts for mineralization as there is overlap between when the Takla volcanics (host to gold-copper mineralization at both Kemess South and Kemess North), and the oldest of the younger Hazelton volcanic rocks were forming. Drilling conducted at Kemess South during the 2019 season had the objective of targeting a chargeability-high feature buried beneath a thick package of Hazelton stratigraphy. Stratigraphic logs were completed from review of historical drill hole logs and 2019 drilling. These logs helped to discern where the historical drilling was stratigraphically through the Hazelton rock package. Supergene alteration and interpretation of paleochannels within the Hazelton rocks should be evaluated for the potential to track the alteration upstream in the paleoenvironment. In addition, at Kemess North, little work has been done to understand the Hazelton stratigraphy. Stratigraphic logs for drill holes through the Kemess Offset target area, Kemess East, below KUG and north of the North Boundary fault may prove useful if stratigraphic logs are used as barcodes to determine stratigraphic offset in adjacent fault blocks.

10 DRILLING

Diamond drilling at the Kemess property totals over 330,000 m of drill core in nearly 900 drill holes, and was designed for mineral exploration, resource delineation and infill, to obtain metallurgical samples, to condemn areas planned for infrastructure, and to gather geotechnical and environmental information. Since 2019, Centerra has completed 27,357 m of diamond drilling in 51 drill holes at Kemess with various purposes including; metallurgical, geotechnical, resource infill and expansion, brownfield exploration, and greenfield exploration (Figure 10-1, Table 10-1, Table 10-2). Most of the drilling samples were collected from NQ (47.6 mm core diameter) diamond drill core.

All drilling campaigns completed on the Kemess property are summarized in Table 10-1 and Table 10-2. Values for mineral-focused exploration, resource infill/expansion, geotechnical and metallurgical drilling are included. In 2020, five drill holes were completed as re-entries of historical Kemess East drill holes, extended past their original final depth and drilled to a deeper length. For the re-entry holes, the original meterage has been included in the year the original holes were drilled, and the extensions completed in 2020 included in the 2020 drill metreage. Figure 10-1 shows all drill hole collar locations for the property, including historical and recent drilling programs.

A detailed description of the Kemess Main Zone historical diamond drilling programs (1987–2016) is included in the previously published Golder Associates Kemess Technical Report, effective date July 12, 2017 (Item 9 – Drilling). Drill programs completed at Kemess Main Zone by AuRico and Centerra (2016-2024) since the previous technical report was published, are described in this section. Historical drilling and assay data (pre-2019) at Kemess South used in the updated 2025 Centerra resource model, are also described in this section.

Figure 10-1: Drill hole collar map Kemess Project

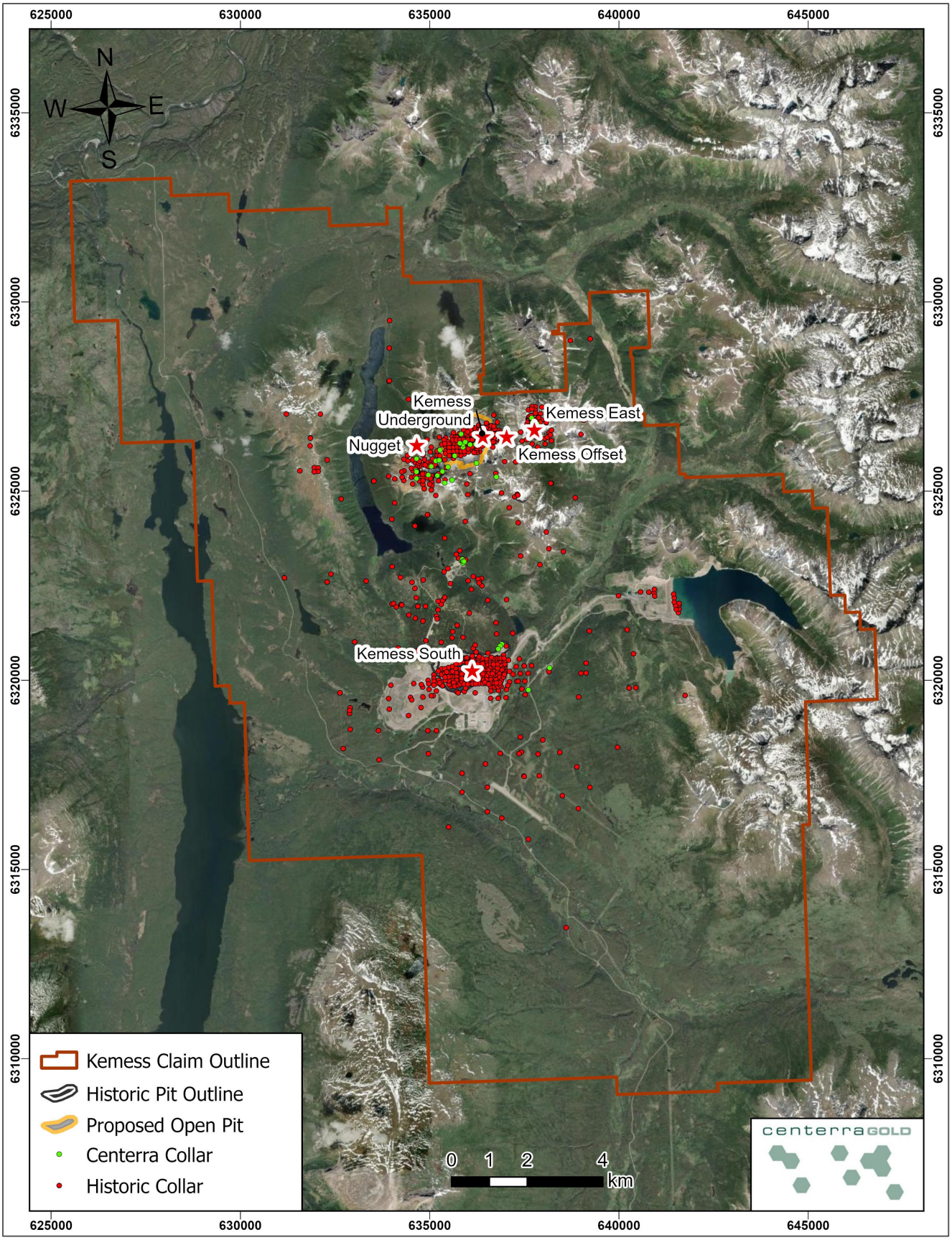


Table 10-1: Drilling programs summarized by year

Year	Metres	Holes	Company
1970	79.30	1	Kenneco Explorations Ltd
1975	588.57	5	Getty Mines Ltd and Shell Oil
1976	1,476.84	8	Getty Mines Ltd and Shell Oil
1977	442.58	4	Kenneco Explorations Ltd
1984	322.82	6	Kenneco Explorations Ltd
1988	870.20	11	El Condor Resources Ltd
1989	731.82	5	El Condor Resources Ltd
1990	6,059.70	34	El Condor Resources Ltd
1991	29,015.23	158	El Condor Resources Ltd
1992	10,504.91	54	El Condor Resources Ltd
1993	1,251.96	19	Kenneco Explorations Ltd
1994	1,955.90	13	Kenneco Explorations Ltd
1999	2,193.65	14	Kenneco Explorations Ltd
2000	7,487.65	36	Northgate Exploration Ltd
2001	8,385.70	16	Northgate Exploration Ltd
2002	33,990.86	62	Northgate Exploration Ltd
2003	24,969.88	77	Northgate Exploration Ltd
2004	19,320.30	79	Northgate Minerals Corp.
2005	15,723.20	40	Northgate Minerals Corp.
2006	11,568.43	42	Northgate Minerals Corp.
2007	18,393.60	30	Northgate Minerals Corp.
2008	1,038.88	7	Northgate Minerals Corp.
2010	16,570.00	34	Northgate Minerals Corp.
2011	6,168.95	19	AuRico Gold Inc.
2013	13,330.82	9	AuRico Gold Inc.
2014	16,872.50	12	AuRico Gold Inc.
2015	29,136.65	36	AuRico Gold Inc.
2016	18,555.5	14	AuRico Gold Inc.
2017	13,940.10	11	AuRico Gold Inc.
2019	8,372.70	9	Centerra Gold
2020	7,558.40	11*	Centerra Gold
2024	11,426.07	31	Centerra Gold
Total	**338,303.67**	**896**	

*5 drill holes are re-entries; the original meterage has been included in the year the original holes were drilled.

Table 10-2: Drilling programs summarized by zone

Zone	Metres	Holes	Years drilled
KUG	90,840	219	1975–present
Kemess East	101,645	86	2002–2020
Kemess Offset	18,575	24	1992–present
Nugget	38,094	89	1976–present
Kemess South	52,163	307	1984–present
Kemess Greenfield Exploration	36,988	171	1970–2015
Total	**338,304**	**896**	

10.1 DATA COLLECTION

10.1.1 Pre-2019 Kemess Main Zone Drilling

A detailed description of data collection for the Kemess historical diamond drilling programs (1987–2016) is included in the previously published Golder Associates Kemess Technical Report, effective date July 12, 2017 (Item 9 – Drilling). The Geology QP has reviewed the drilling database and is of the opinion that it is sufficient for use in resource estimation.

Two drill programs completed by AuRico in 2016–2017 at Kemess East, were not described in the previous 2017 technical report and are included in this report and resource model update.

The 2016 exploration program spanned from June 18 to October 14 and totalled 18,555.5 m in 13 drill holes targeting the Kemess East deposit. Drilling was completed by Driftwood Diamond Drilling of Smithers. Three drills were used on site, two Zinex A5 machines and one CPI-3000. Drill sites were accessible by road and supported by helicopter. Drills were collared with HQ drilling down to an average depth of 507 m and then reduced to NQ for the bottom of the hole. All possible NQ drill core was oriented using the 2ic EzyMark supplied by Reflex Instruments.

The 2017 exploration drill program spanned from June 26 to October 30. Drilling totalled 13,930.1 m in 10 drill holes. The 2017 drilling was completed by Driftwood Diamond Drilling of Smithers. Two drills were used on site, one Zinex A5 machine (A5-2) and one CPI-3000 (CPI). Drill sites were accessible by road and supported by helicopter. Drills were collared using HQ rods down to an approximate depth of 500 m and then reduced to NQ for the remainder of the hole.

All possible NQ drill core was oriented using the Reflex Ori-BlockTM, for holes drilled by the CPI, and by the Reflex ACT IIITM system, for holes drilled by the A5-2. The ASD TerraSpecTM Halo mineral identifier, a NIR spectrometer, was used to analyse the matrix of the rock for all samples logged after August 3, 2017. Readings were taken on dry core of each sample with minimal to no veining at an opportune location within the sampled interval and recorded as point data in the database. The sample area was circled so the locations can be referenced from the core photos.

For both 2016 and 2017 drill programs, all drill holes were logged by on site AuRico geologists for lithological, mineralogical, structural and geotechnical properties including the collection of detailed structural data on joints and vein sets when core was orientable. Sample intervals were determined by the logging geologist and ranged from about 0.3 m to 2.5 m in length for all core sizes. Sample intervals were chosen so as to not cross lithological or significant alteration boundaries. Magnetic susceptibility was measured every metre and collected as point data downhole, and specific gravity measurements were collected approximately every 10 m, or five samples downhole. All the drill core was digitally

photographed including complete box photographs as well as detailed unit, mineralization and alteration photographs.

10.1.2 2019-2024 Kemess Main Zone Drilling (Centerra)

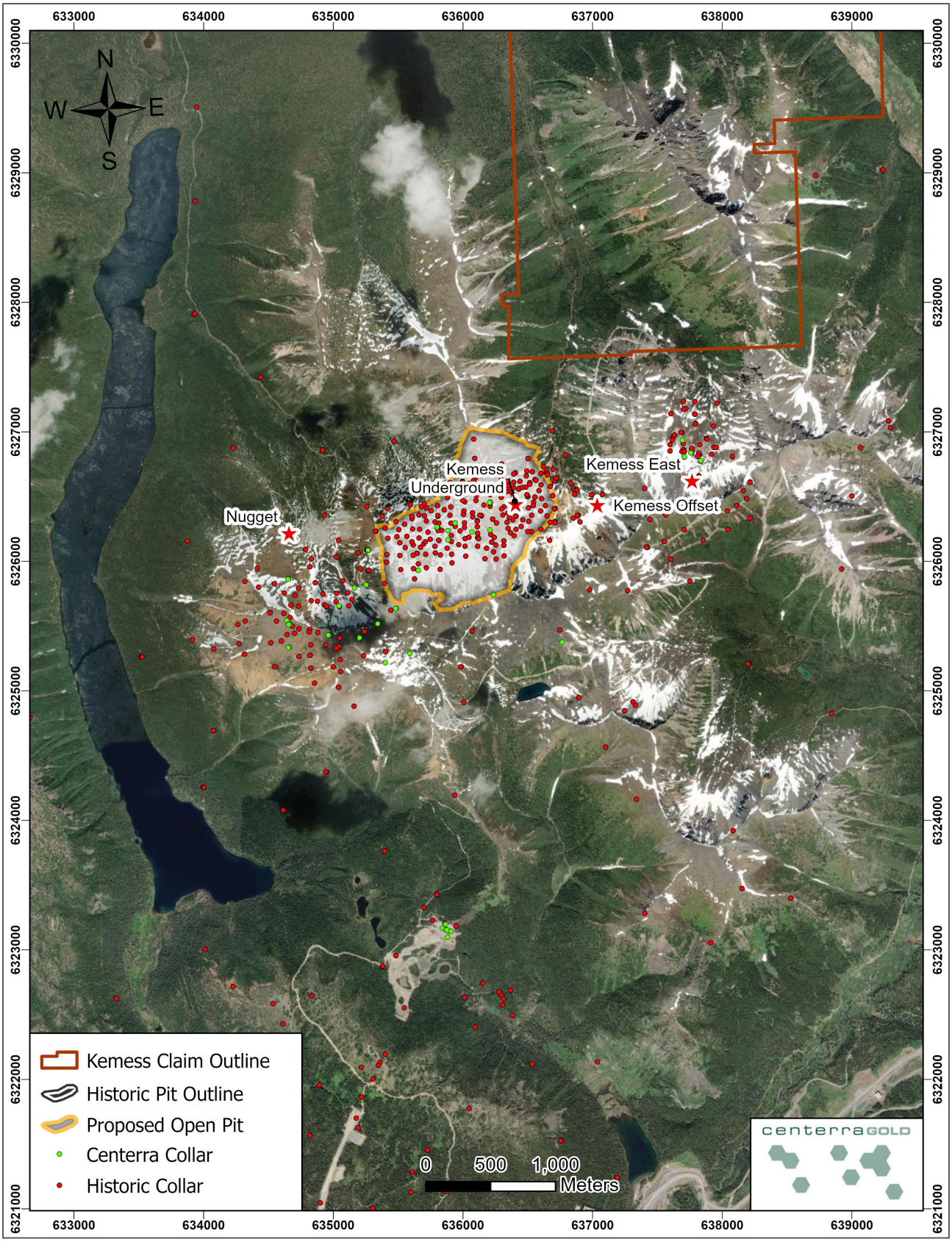
Since 2019, Centerra has completed a total of 25,747 m of diamond drilling at the Kemess Main Zone project area including the Nugget, KUG, Kemess East, and KOZ zones (Figure 10-2). Project management of mineral-focused drilling programs in 2019 was conducted by Equity Exploration Consultants out of Vancouver, British Columbia, and by Centerra geologists for 2020–2024 drill programs.

In total, 6,452.3 m was drilled in 2019 along the Kemess Main mineralization trend and an additional 1,610.0 m of drilling was completed east of Kemess South. The exploration diamond drilling was completed in two phases. Phase I drilling focused on exploration targets related to the Nugget and KUG deposits at Kemess Main and took place from June 4 to August 1. Phase II drilling focused on an exploration target located east of the Kemess South pit and took place from September 17 to October 8.

Centerra resumed exploration drilling in 2020 from June 25 to September 29 at the Kemess Main Zone area. In total, 7,558.4 m of drilling was completed at the Nugget and Kemess East zones. Five of the seven holes completed at Kemess East were re-entry extensions of pre-existing drill holes, to test for extensions of mineralization at depth. Drilling at Kemess East confirmed an extension of the Kemess East deposit, referred to as the Kemess East Deep zone.

In 2024, Centerra re-assessed the historical Kemess data and constructed updated 3D geological and resource models. Internal technical studies commenced to assess different potential mining scenarios. Exploration and infill drilling programs resumed, completing a total of 11,423 m of diamond drilling at the Nugget and KUG areas. The objectives of the program included resource infill and expansion of known near-surface mineralization at Kemess Main Zone to upgrade the overall Kemess inventory, to better define the ‘Broken Zone’ at the Main Zone, and to take samples for geotechnical and metallurgical characterization.

Figure 10-2: Drill hole collar map Kemess North



All 2019 to 2024 drilling at the Kemess property was conducted by Driftwood Diamond Drilling based out of Smithers, British Columbia, using Zinex A5 (A5) and CPI 3000 (CPI) drill rigs. Servicing of the drills, including transport of crews at crew change, slinging of fuel, water pumps, core boxes, and drilled core back to the Kemess mine site, was largely completed using a AS350 B2 helicopter provided by Silver King Helicopters out of Smithers, British Columbia.

Detailed core logging and sampling was done by Equity in 2019, and by Centerra-Terralogic Exploration Ltd of Cranbrook, British Columbia, in 2020–2024. For 2019–2024, all drill holes were logged by onsite geologists for lithological, mineralogical, structural, and geotechnical properties.

Geotechnical logging included, Total core recovery was measured for all drill holes to the nearest centimetre for each drill run from the base of overburden to the end of hole. Rock quality designation (RQD) was measured for every run between overburden to end of hole. The RQD measurement recorded was calculated as the sum of all naturally fractured core lengths >10 cm in length divided by the measured total core recovery value. Natural fractures include breaks that are inherent to the lithology (e.g. joints), but not mechanical breaks generated by man-made breaks. Diametric point load tests were conducted on all 2019–2024 drill core for the drilling programs. Measurements were taken at 10 m intervals from the base of overburden to end of hole with a Roctest PIL-7 point load tester. Samples selected were greater than one core diameter in length. Ideally, samples were between 10 cm and 16 cm.

Magnetic susceptibility was measured on all drill core. Measurements were collected every metre from the base of overburden to the end of hole. Measurements were taken using a Terraplus KT-10 magnetic susceptibility meter as well as using an MPP-EM2S+ multi-parameter probe (MPP) from Instrumentation GDD. Specific gravity (SG) measurements were collected on all drill core. Measurements were collected every 10 m from the base of overburden to the end of hole. On average, samples selected for SG measurements were 10 cm long. SG values were calculated by dividing the dry mass of a sample by its wet mass. The wet mass was measured by measuring the mass of the rock sample suspended in water.

All drill core was digitally photographed including complete box photographs, and detailed unit, mineralization, and alteration photographs. Photo details for each photo, included the drill hole name, the core photo interval in metres and the box numbers of the boxes present in the photograph. Both wet and dry photos were captured for the drilling programs.

Geological data collection from 2019–2024 was conducted using acQuire 4 logging software. Data fields collected in acQuire included lithology, mineralization, alteration, veining and structure, entered in separate data tables within the acQuire program. Lithological logging involves a description of the rock including its colour, grain size, texture (e.g. equigranular, aphanitic, porphyritic), mineralogy, structures (e.g. bedding, veins, foliation, etc.), variation and nature of contacts (if observable). Alteration minerals

and their intensities was recorded in acQuire, Alteration minerals logged at Kemess include quartz, k-feldspar, biotite, chlorite, epidote, albite, magnetite, sericite, calcite and zeolite. The vein classification scheme from Kemess North developed by McKinley (2006) was used as the foundation of the vein logging scheme. Veins were classified on descriptive criteria including morphology, texture, mineralogy of vein filling, alteration envelopes and orientation ordered in decreasing age. Mineralization data was recorded for the full hole length and consists of metallic minerals (chalcopyrite, pyrite, galena, sphalerite, etc.). Species, form and abundance of mineralization was recorded in acQuire. Logging geologists recorded structural range features, such as sheared, broken, rubbly, faults, etc. as intervals, whereas point features such as bedding, foliation, jointing, etc. were also recorded. An intensity was given to each structure recorded from trace (1) to intense (5). Alpha angles (angle to the core axis), if measurable, were also recorded. Beta angles were not recorded as the core was not oriented.

Terraspec® short-wave infrared (SWIR) and near infrared (NIR) spectral analyses were collected on all drill core for both whole core and cut core from the 2019–2024 drilling programs. Spectra were collected at 1–3 m intervals from the base of overburden to end of hole. In addition to the spectral capture, sample location depths were measured, marked with a grease pencil on the core, described, and photographed. Portable x-ray florescence (pXRF) analyses were collected on drill core for select portions of the 2019–2024 drill core.

In 2024, the Centerra database management team, validated and imported historical data to the Kemess acQuire database. In this database review, several unclear items were addressed including the historical Kemess Mine Grid. All historical Kemess engineering data has been designed in the local ground coordinate system (e.g. block models, Kemess South pit, Kemess South infrastructure, KUG infrastructure).

A difference of ~1.87° shift in between the Kemess Mine Grid (aligned with True North) and NAD83 UTM Zone 9 at the Kemess property. When using the Kemess Mine Grid X,Y,Z coordinates the Azimuth_Kemess_MG value should be used in the Survey table.

Multiple different transforms between NAD83 UTM Z09N and the mine grid have been created. The drilling database has utilized the following 2001 transform for consistency despite newer ones surveyed in 2007 and 2017 based around the Kemess South pit and tailings storage facility (TSF).

To convert from Mine Grid to UTMs:

- Scale: 0.9997655 Rotate: -1.872777 Shift: N +6309986.944 E +626479.931

To convert UTMs to Mine Grid:

- Scale: 1/0.9997655 Rotate: +1.872777 Shift: N -6309986.944 E -626479.931

For the Centerra geological drilling and surface sampling, a two-point transform was created from the above transformation using the two points detailed in Table 10-3.

Table 10-3: Transformation coordinates used for Kemess Mine Grid to UTM conversion

Point	Local east	Local north	Local elevation	Base east	Base north	Base elevation
1	11864.946	16776.133	1651.107	637787.638	6327137.843	1651.107
2	12897.335	17743.145	1317.340	638787.638	6328137.843	1317.340

10.1.3 Pre-2019 Kemess South Drilling

From 1984 to 2019, 52,163 m in 307 diamond drill holes have been completed at the Kemess South deposit (Figure 10-3). The drill spacing was completed on 50 m sections with 100 m spacing between holes on a section. The subsequent section of drill holes is offset 50 m in a north-south direction from the previous section of drill holes. The configuration creates five-point spacing. Britton Brothers Diamond Drilling Ltd was the sole drill company used to complete the holes drilled between 1996 and 2004. Suisse Diamond Drilling completed the 2006 program.

For historical drill programs from 1984 to 1996, drill core was delivered to the core logging building at the Kemess Mine complex, where a geologist completed a written log(s) for the drill hole that includes geological and geotechnical information. All drilling, logging, and sampling at the Kemess Mine for that period was in the S.I. system of measurement.

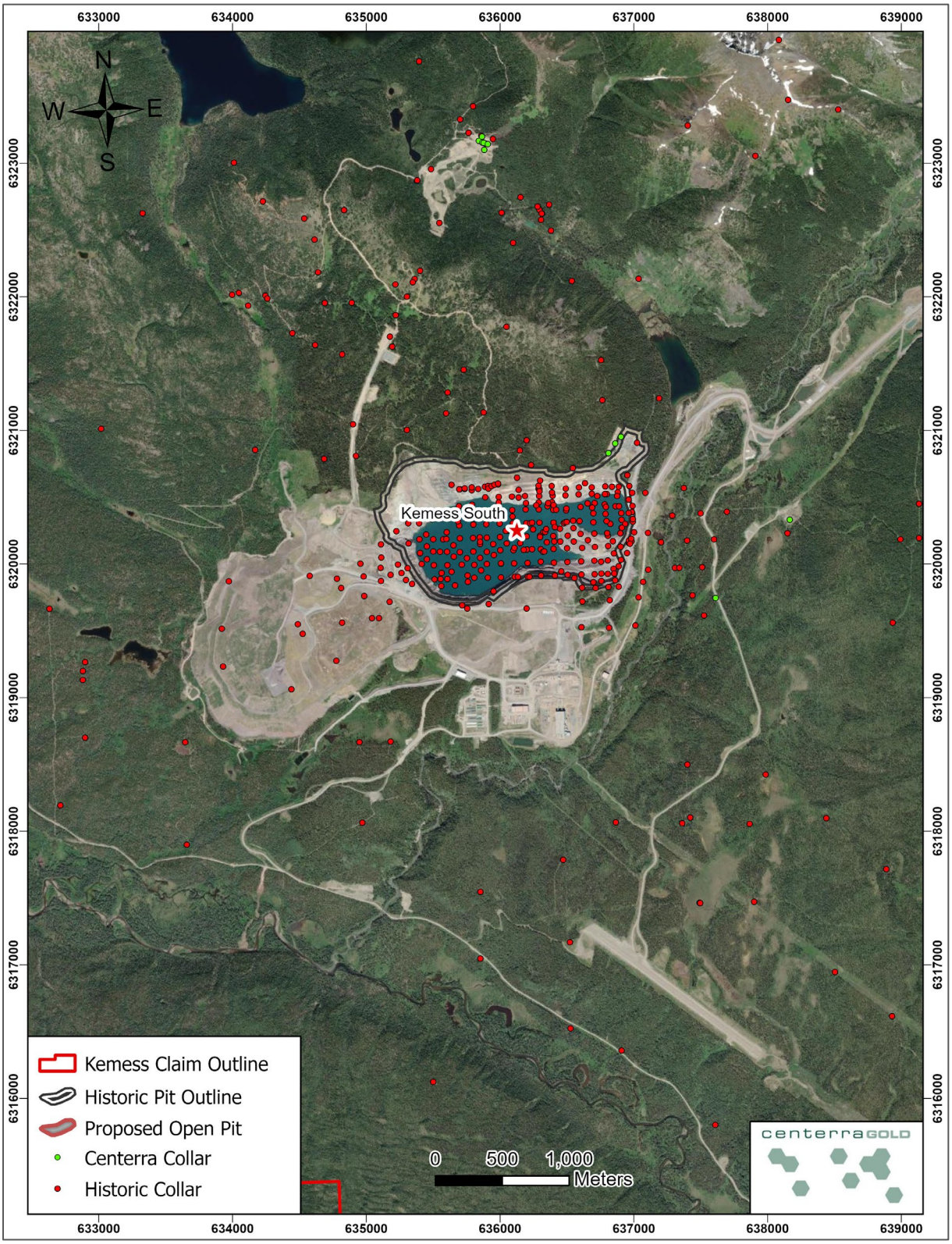
The geological data included identification of specific geological formations, colour, presence and visual estimate of sulphide minerals, nature of fracture filling and veins, and a detailed geological description of the core that includes textural and lithological characteristics, deformation, contact styles and mineralization.

Geotechnical data were also recorded, including core recovery, RQD, joint condition, weathering, and hardness. Core recovery was calculated by cumulatively measuring the pieces of core between two footage markers in the core box. This measure link is then described as a percentage of the actual core length as indicated by the footage markers. RQD measurements are captured for pieces of core that are greater than twice the core diameter. For HQ drilling the core pieces must be greater than 7.6 cm long and for NQ drilling, greater than 10.2 cm long. RQD is calculated as the percentage of all core pieces greater in length than the minimum requirement for each 2 m length of the drill hole. Fracture intensity is measured by counting the number of open fracturing in core competency for every 20 cm of core. SG was measured at varying intervals in mineralized sections. A 20 cm long sample was selected, measured and weighed on site using the wax immersion method to provide a specific gravity calculation.

Drill hole data was digitally recorded by Northgate Mineral employees, starting in the 2000s, at the mine site into an access database to be collated with collar and downhole survey data as well as assay data.

Completed hole logs were printed and checked by project geologists or supervisors to ensure their completeness and accuracy.

Figure 10-3: Drill hole collar map Kemess South



10.2 SUMMARY AND INTERPRETATION OF 2019–2024 DRILLING

10.2.1 Kemess Main Zone Drilling

The 2019–2024 drilling by Centerra has continued to define the Kemess Main Zone trend, an east-westerly striking prospective belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east, the trend includes the Nugget, KUG, Kemess Offset, Kemess East, and Hilda South targets, with a general trend of deposits occurring at greater depths towards the east (Figure 10-3).

The zones that make-up the Kemess Main trend are summarized below from west to east, with additional details described in Item 9 (Exploration). For each zone, selected significant composite assay intervals are provided. The results are composited using the following criteria, composite assay intervals must be longer than 2.0 m, have grade greater than 0.1 g/t Au or 1% Cu and include maximum internal waste of 4.0 m where it exists.

The 2024 drill results show shallow mineralization at the Kemess Main Zone, with potential for future resource addition and exploration. The 2024 exploration drilling at Nugget continued to expand low-grade shallow porphyry mineralization, with varying Au:Cu ratios, typically high gold to low copper volcanic hosted, with an increasing Au:Cu ratio moving west. Significant drill intercepts include:

- Drill hole KN-24-018: 282.0 m @ 0.271 g/t Au, 0.100% Cu from 10.0 m
- Drill hole KN-24-024: 120.5 m @ 0.267 g/t Au, 0.137% Cu from 3.3 m
- Drill hole KN-24-029: 362.0 m @ 0.255 g/t Au, 0.057% Cu from 135.0 m.

Results from 2024 infill drilling at Kemess Main Zone show wide intervals of significant mineralization on the western margins of the proposed open pit margins. In this portion of the deposit, the upper ~300 m comprises phyllic altered volcanic rocks with underlying potassically altered volcanics, both domains host copper-gold mineralization. Significant drill intercepts include:

- Drill hole KN-24-025: 402.0 m @ 0.423 g/t Au, 0.201% Cu from 47.0 m.

The KOZ between Kemess East and KUG should be drilled to identify any significant mineralization that exists in the fault block between the two deposits (Figure 7-3, Figure 9-3). If there is a mineable body in the Offset zone, it would increase the likelihood of Kemess East being exploited in the same manner as KUG. The 2024 geophysical surveys and 3D model updates show potential for buried porphyry mineralization to the south at KOZ, and potential for mineralization to shallow up-dip of modelled intrusions to the north, in the Takla volcanics at the top of the modeled Black Lake intrusion. The KOZ exploration model is based on similar geometry at the KUG deposit, where the highest-grade gold-copper mineralization is situated at the top of the Black Lake porphyry intrusion (Figure 7-3).

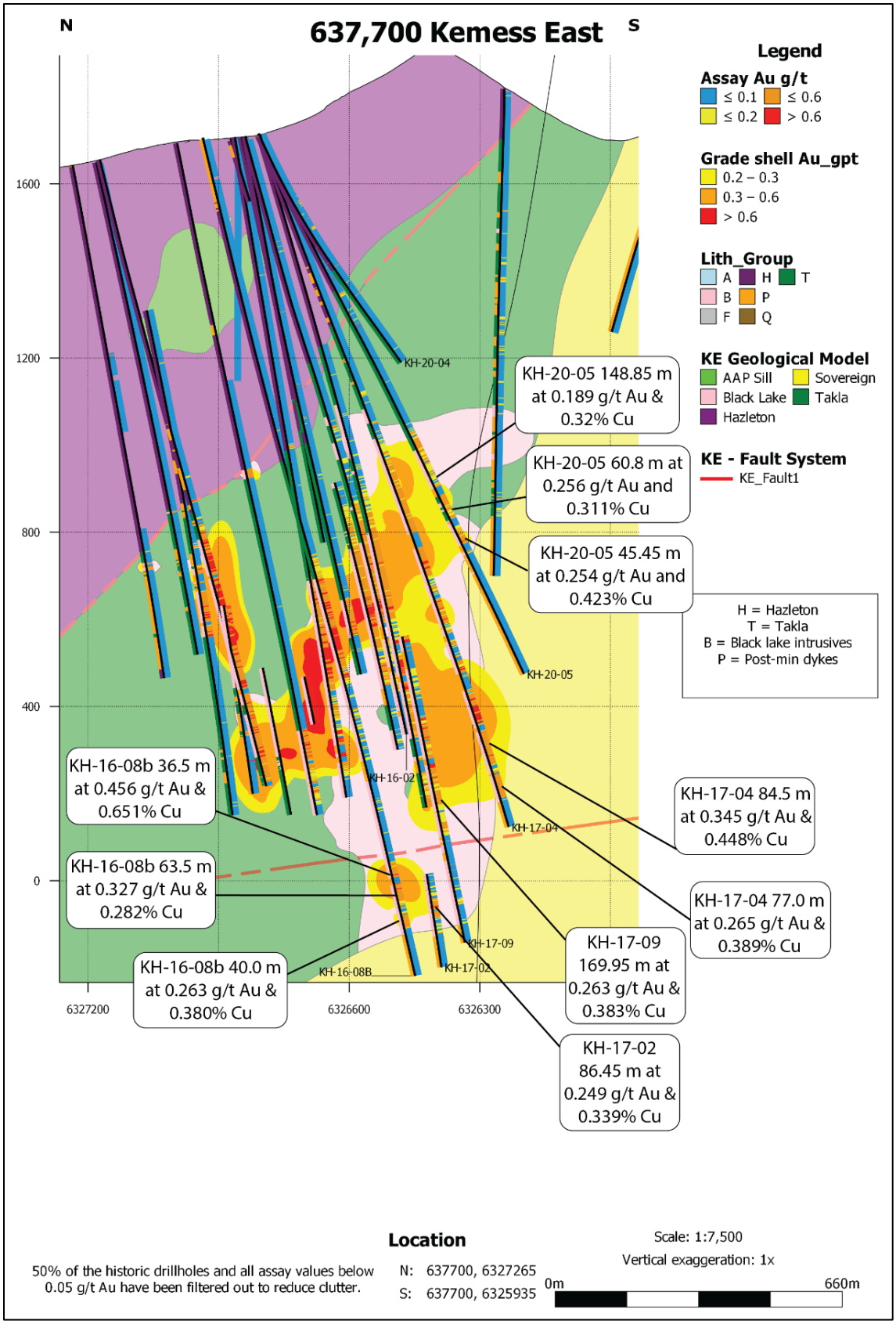
The 2020 drilling at Kemess East confirmed an extension of the Kemess East deposit at depth. The extension is referred to as the Kemess East Deep zone. The Kemess East Deep zone is 550 m long, plunging steeply to the north with a thickness of approximately 200 m, and an east-west width of approximately 250 m. The Kemess East Deep zone is interpreted to be open to the east and down dip. Significant drill intercepts are shown in Figure 10-4 and highlights include:

- Drill hole KH-17-09-extension: 162.6 m at 0.239 g/t Au, 0.380% Cu from 1,509.7 m.

Recent drilling at Kemess East confirmed that the Kemess East deposit remains open to the south by confirming mineralization in a previously untested area that was modeled as the post-mineral Sovereign pluton. Further drilling is needed to test the mineral potential of this zone and fully define the extents of the deposit.

Additional drilling at Kemess East should focus on further defining the location of the Kemess East Offset Fault that bounds the eastern edge of the deposit as well as refining the location of the thrust fault on the deposits northern edge. Locating these faults more precisely may add to the resource potential to both the north and east, holds potential to contain a faulted-off portion of the Kemess East deposit itself, as well as increase model certainty in the deposit. Step-out brownfield exploration drilling east of Kemess East is recommended, including testing of the KEY target (the potential Kemess East eastern extension), and Hilda South target, outlined in Item 9 – Exploration (Figure 9-3).

Figure 10-4: Geological and gold mineralization cross-section at Kemess East



10.2.2 Kemess South Drilling

The 2019 relog and drilling at the Kemess South “Eastern Extension” target, show limited exploration potential for significant gold-copper mineralization to be preserved in the lithologies east of the historical Kemess South open pit.

The drilling data and 2025 updated resource model at Kemess South show potential for additional shallow and deep mineralization west of and below the historical pit (Figure 7-4). The northern limits of mineralization appear to be cut off by the northern, moderately south dipping normal fault that juxtaposes older, unmineralized Asitka Group rocks in the footwall against the mineralized Takla and Black Lake intrusions in the hangingwall, which is not well defined in the western area. Exploration potential exists in both the hanging and footwall of the West Fault: an east-dipping normal fault. The Kemess South open pit mineralization remains open down dip to the west of the existing block model until terminating on the West Fault, exact location unknown due to lack of drill data (Figure 7-4). On the western side of the West Fault, historical drill holes returned shallow gold-copper mineralization in Black Lake intrusions and Takla volcanic rocks, open to the south and west, and along the West Fault. Future drilling is recommended west of the Kemess South open pit in the West Fault area to target potential resource expansion both at surface and at depth.

10.3 SURVEY CONTROL

10.3.1 Pre-2019 Kemess Main Zone Drilling

Pre-2016 surveys are described in the Kemess North Technical Report by Golder Associates (2017), excluding the most recent AuRico Kemess East drill programs in 2016–2017 that are described below. For 2016 AuRico drill programs at Kemess East, all drill hole collar locations were surveyed with Trimble GPS with RTK base station. Collar azimuths and dips were surveyed using a Reflex TN-14 Gyrocompass to measure true north azimuth independent of the significant magnetic field variations on the property. For the 2017 AuRico drill programs at Kemess North, all drill hole collar locations were surveyed by McElhanney Surveying of Smithers on October 12, 2017. Collar azimuths and dips were surveyed using a Reflex TN-14 Gyrocompass. Due to the strong magnetics on the property, particularly within the Hazelton rocks, all collar alignments were done using the Reflex TN14 Gyrocompass™.

10.3.2 2019–2024 Kemess Main Zone Drilling (Centerra)

In 2019, the azimuth of the drill was aligned using a north-seeking downhole tool, aided by the foresights and back sights laid out by the exploration team. The gyroscopic compass used for rig alignments was called the Champ Gyro and was supplied by Survey-Tech, based in Ontario, Canada. The Champ Gyro is a non-magnetic compass, unaffected by magnetic interference. It produces true-north azimuth measurements to within ±0.75° and inclination measurements to within ±0.15°. The dip was either set

using an inclinometer measured against the piston shaft of the drill head or by using the Champ Gyro placed inclined, nose-down on the ground, resting within the chuck-jaws of the drill mast.

The 2019 drill collar surveys were completed by a professional surveyor (Jeff Johnstone) from the Kemess mine department using a differential GPS and base station. A Trimble R10-2 GPS system with a 35-watt external base radio antenna was used to collect the collar locations. The shot accuracy with this unit is 3–10 mm horizontal and 6–15 mm vertical. Survey stations were taken beside and up against the drill pipe sticking out of the ground. Survey stations were recorded on the Trimble data collector and than exported data out as a CSV, then provided to Equity and Centerra staff.

For 2020, drill collar surveys were completed by professional surveyors (Austin Corcoran and Josh Piercey) from the Kemess mine department using a differential GPS and base station. A Trimble R10-2 GPS system with a 35-watt external base radio antenna was used to collect the collar locations. The shot accuracy with this unit is 3–10 mm horizontal and 6–15 mm vertical. Survey stations were taken beside and up against the drill pipe sticking out of the ground. Survey stations were recorded on the Trimble data collector and than exported data out as a CSV.

For 2024, drill collar surveys were completed by professional surveyor (Ian Bissonnette) from HBH Land Surveying of Smithers BC using a differential GPS and base station. Equipment used for this survey included a Spectra Precision SP85 RTK Rover receiver with a spec of 8 mm + 1 mm PPM H and 15 mm + 1 mm PPM V while running a Spectra Precision SP80 Base station that has similar specs. The base station uses a 2-watt radio and a 35-watt repeater radio was also used to get signal down over the crest of the mountain. The new base station setup (HBH100) was established by running a simultaneous static observation to the Kemess base station Trimble R10 at point 3002 (Static spec 3 mm + 0.5 PPM H and 5 mm + 0.5 PPM V for SP80 and R10).

10.3.3 Pre-2019 Kemess South Drilling

For historical drilling at Kemess South, including programs managed by Northgate, Kemess staff surveyors were used to complete the final collar survey prior to the rig moving off the set-up. Kemess mine grid co-ordinates were used and data recorded in handwritten logs, or for 2004 programs onwards, in the access database.

10.4 DOWNHOLE SURVEYS

10.4.1 Pre-2019 Kemess Main Zone Drilling

Pre-2016 surveys are described in the Kemess North Technical Report by Golder Associates (2017), excluding the most recent AuRico Kemess East drill programs in 2016–2017 that are described below. Due to the often very strong magnetics on the property, particularly within the Hazelton rocks, all holes drilled during 2016 were surveyed with the Reflex® Gyro non-magnetic downhole survey instrument.

Survey points were taken every 10 m from bottom to the top of each hole. At least two surveys were completed on each hole, one approximately 100 m after reducing to NQ core and the second at the bottom of hole.

For 2017 drill programs, downhole surveys were completed using the non-magnetic Reflex EZ-GyroTM. Survey points were taken every 50 m from bottom to the top of each hole. At least two surveys were completed on each hole, one approximately 100 m after reducing to NQ core and the second at the bottom of hole. Two to three overlapping points were taken to ensure accuracy and continuity between the two surveys.

10.4.2 2019–2024 Kemess Main Zone Drilling (Centerra)

For 2019 drill holes, downhole surveys were completed using the Champ Gyro from Survey-Tech. The Champ Gyro is a gyrocompass capable of capturing single-shot (low precision) and multi-shot (high-precision) point surveys as well as multi-shot continuous surveys, which allow for multiple shots over a period of time while the tool is being lifted or lowered within the borehole. Single shot surveys were completed during the drilling process and were collected at 50 m intervals starting at 50 m from the top of the hole. Continuous surveys were completed after the drilling process was terminated. Continuous surveys were generally run from surface at 0 m depth towards the end of hole. This was called an “in” survey. A second survey would follow and be taken from the bottom of the bore hole to surface at 0 m depth. This second survey was called an “out” survey. A continuous survey was comprised of spot tests taken at 5 m intervals. Generally, the single shot survey data is superseded by the continuous survey data.

In 2020–2024 programs, downhole surveys were completed using the Sprint IQ continuous gyro tool from Reflex. The Sprint IQ is a gyrocompass capable of capturing multi-shot point surveys as well as multi-shot continuous surveys, which allow for multiple shots over a period of time while the tool is being lifted or lowered within the borehole. Continuous surveys were completed after the drilling process was terminated. Where practical, short surveys were conducted after casing and again at approximately 200 m depth, to check for excessive deviation. Continuous surveys were generally run from surface at 0 m depth towards the end of hole. This was called an “in” survey. A second survey would follow and be taken from the bottom of the bore hole to surface at 0 m depth. This second survey was called an “out” survey. A continuous survey was comprised of spot tests taken at 10 m intervals. In cases where both ‘in’ and ‘out’ measurements were taken, both were stored in the database, but only one would be approved and used in Mineral Resource estimation.

In 2024, the Centerra database management team, validated and imported historical data to the Kemess acQuire database. In this database review, several unclear items were addressed including the historical downhole survey data. For downhole survey classification, five different categories were created and

recorded for quality assurance data including measured, calculated, adjusted, estimated and inferred. Any data recorded to the database based on the original readings from instrument raw data is classified as ‘measured’. For the ‘adjusted’ category, adjustments have been made either between UTM and local mine grid or adjustments must be applied where conversions have been made between magnetic north and true north values. ‘Estimations’ occur where Flexit readings exist, but no recorded data is available and occur every 100 m.

10.4.3 Pre-2019 Kemess South Drilling

For historical drilling at Kemess South, downhole surveys were completed with Sperry-Sun and Flex-IT survey tools. Majority of drill holes at Kemess South are of vertical or near vertical dip. The type of downhole survey tool used is recorded for each drill campaign when available, in the 2025 updated Centerra Kemess acQuire database.

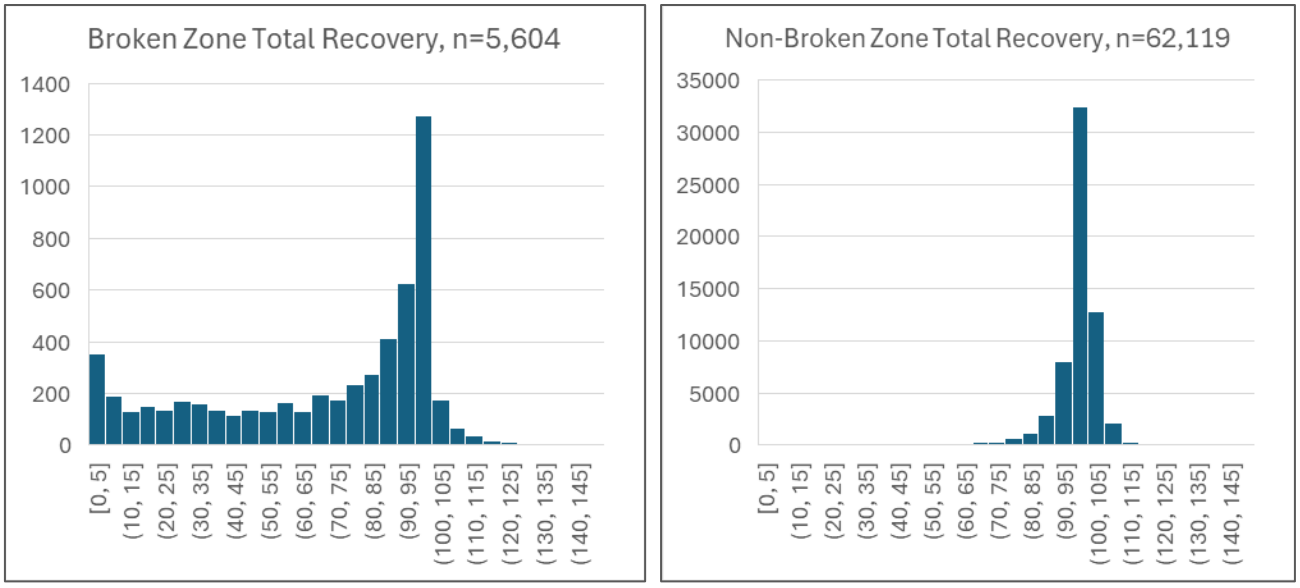
10.5 CORE RECOVERY

The ‘Broken Zone’, within the Kemess Main Zone, presents challenging drilling conditions. Historically, drilling an HQ diameter hole (64 mm core) to act as a casing for NQ (48 mm core), was used to complete the hole, solving the problem. In rare instances reduction to BQ (37 mm core) was necessary to reach target depth. The core recovery is lower in the Broken Zone than in the surrounding rock, with an average of ~70% within the Broken Zone and approximately 97% in the rest of the Kemess Main deposits, including Kemess East.

10.5.1 Pre-2019 Kemess Main Zone Drilling

For drill core from pre-2019 drill programs at Kemess Main Zone, within the Broken Zone, the total core recovery averaged 67.9%. Drilling from the same period outside of the Broken Zone at KUG averaged 96.7% total core recovery (Figure 10-5).

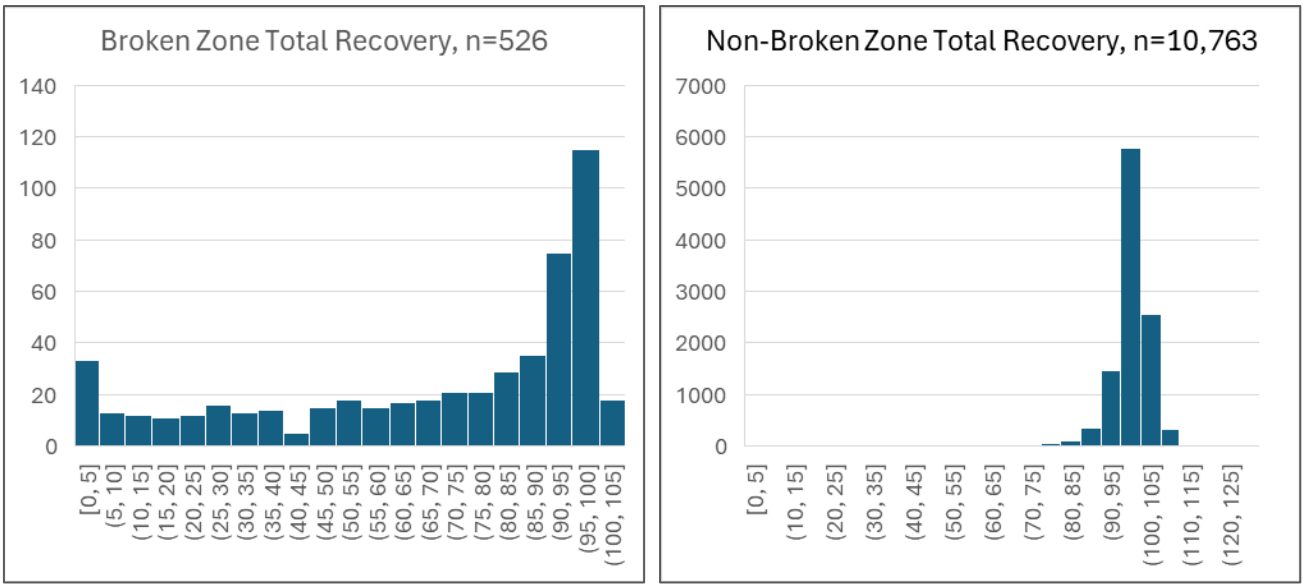
Figure 10-5: Histograms plotting total core recovery (%) for Kemess Main Zone from historical pre-2019 drilling



10.5.2 2019–2024 Kemess Main Zone Drilling (Centerra)

For drilling conducted by Centerra between 2019 and 2024 within the Broken Zone, drill core averaged 68.3% total core recovery. For drilling from the same period outside of the Broken Zone, drill core averaged 97.8% total core recovery (Figure 10-6).

Figure 10-6: Histograms plotting total core recovery (%) for Kemess Main Zone from Centerra drilling, 2019–2024

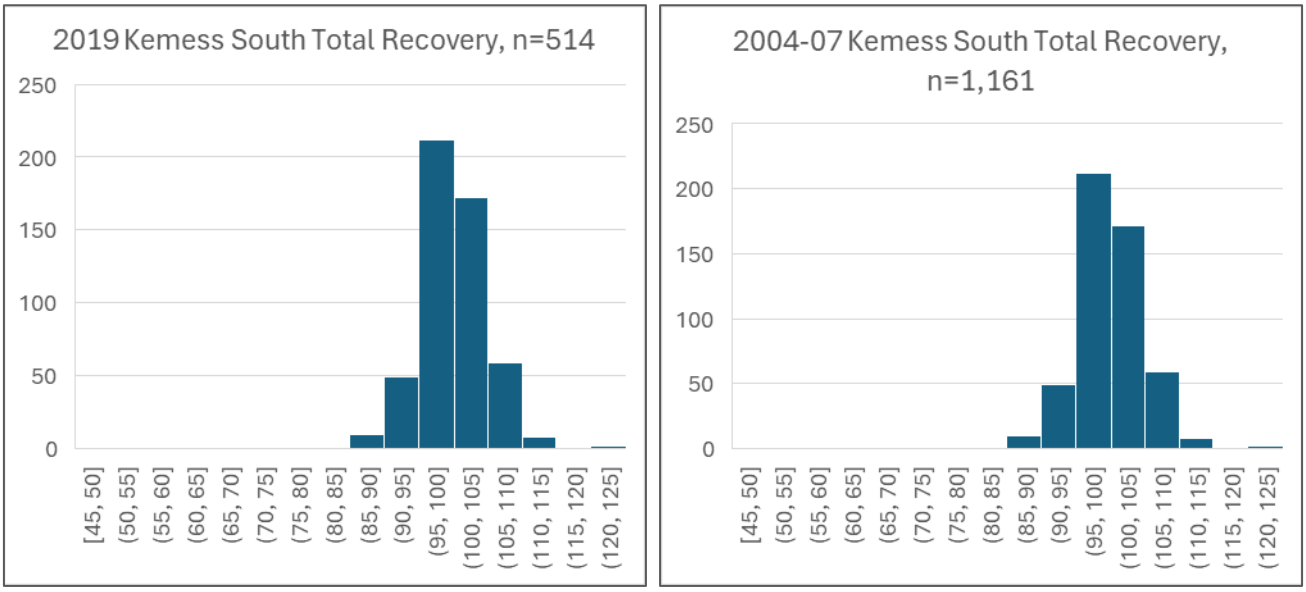


10.5.3 Kemess South Drilling

Recovery data from drilling campaigns prior 2004 were not preserved. Total core recovery collected between 2005 and 2007 averaged 92.4% for Kemess South. For the 2019 drill program targeting the

Kemess South eastern extension, ~2 km east of the historical open pit, the core recovery was 100.4% (Figure 10-7).

Figure 10-7: Histograms plotting total core recovery (%) for Kemess South (2004–2007) drilling and Centerra 2019 drilling



11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 PRE-2019 SAMPLES – KEMESS MAIN ZONE

Pre-2016 drill samples are described in the Kemess North Technical Report by Golder Associates (2017), excluding the most recent AuRico Kemess East drill programs in 2016–2017 that are described below.

11.1.1 Sampling Method and Approach

Sample intervals were determined by the logging geologist and ranged from about 0.3 m to 2.5 m in length for all core sizes. Sample intervals were chosen so as to not cross lithological or significant alteration boundaries.

11.1.2 Sample Preparation and Laboratory

For 2016–2017 samples, diamond drill core samples were split or sawn, crushed to 80% minus 10 mesh and, when equipment allowed, pulverized to better than 85% minus 150-mesh (-75 µm) at an on-site sample preparation laboratory. When pulverisation was not possible on site it was completed at ALS Chemex on AuRico's behalf instead. Operation of the sample preparation laboratory and the quality control procedures were implemented under the supervision of J. Wade Barnes, Professional Geoscientist and onsite QP.

The prepared samples, weighing approximately 200–210 g, were analysed by a pXRF on site for real-time monitoring of geochemistry and then submitted to ALS Chemex in North Vancouver for final assay analysis.

11.1.3 Assaying

For 2016 and 2017 programs, ALS lab, pulverized samples to 85% passing 150 mesh when needed. Samples were analyzed for a suite of 33 elements in 2016 and 48 elements in 2017, including iron, molybdenum, and silver, using four-acid digestion and inductively coupled plasma – atomic emission spectroscopy (ICP-AES) on a 1 g subsample. Significantly mineralized samples (as determined by on site XRF analysis of crushed and pulverized drill core) were additionally analyzed by an extra ore grade analysis for copper and molybdenum by ICP-AES, following a four-acid digestion. Gold analyses were completed by standard 30 g fire assay with an AA finish.

ALS Minerals laboratories are accredited ISO 9001-2008 by QMI and the North Vancouver Laboratory is accredited ISO 17025-2005 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay samples submitted by AuRico Metals. ALS also

participate in a number of international proficiency tests, such as those managed by CANMET and Geostats.

11.1.4 Quality Control

For 2016 and 2017 programs, quality control samples (blanks, duplicates, and standards) were inserted into the sample stream at regular intervals such that 2 in 25 samples were submitted for quality control purposes. Additional quality control samples were inserted into the sample stream at the geologist's discretion. The total percentage of all samples sent to lab QA/QC samples 8–9% of the entire sample population submitted to ALS Chemex.

11.1.5 Security

Sample cutting, pulverizing, and sample shipment preparation was managed on site by AuRico supervisors. For 2016–2017 programs, the half drill core and crushed rejects (80% -10 mesh) are retained at site along with the remaining core in the core storage at "Lower Camp" near Kemess Lake, located at approximately 637,540E and 6,321,280N (NAD83 UTM Zone 9).

11.2 2019–2024 CENTERRA SAMPLES – KEMESS MAIN ZONE

11.2.1 Sampling Method and Approach

Diamond drill core samples were collected from predominantly NQ diameter diamond drill core. HQ diameter (63.5 mm) drill core was also sampled when logged. Sample intervals were predominantly of 2 m core length but were shortened at the discretion of the geologist at lithological, structural, or major alteration contacts. Geologists identified all core samples with a unique sample ID number and marked all sample intervals with sample tags in the core box. Prior to detailed logging and marking of the sample intervals, technicians logged the core for geotechnical characteristics. After the geologist logged the core in detail, the drill core was marked with the sample intervals, cut lines, and assigned sample numbers. The wet and dry drill core was photographed prior to being sent for cutting.

The NQ and HQ drill core was cut in half using a diamond blade on a closed circulation (sump with overflow tank and pump assembly) electric core saw, with one half placed in a sample bag for shipping to the laboratory and the other half placed back in the core box for future reference. Cut core was sampled from the same side consistently.

11.2.2 Sample Preparation and Laboratory

From 2019 to present, drill samples were cut at the project site and half-core samples were sent for sample preparation and pulverization to BV in Vancouver, British Columbia.

A subset of 3.8% of samples from 2019–2024 drill core samples were sent to SGS Laboratories (SGS) in Vancouver, British Columbia, as an independent check for analytical bias and accuracy. The SGS laboratory in Vancouver, British Columbia is accredited by the SCC for specific tests listed in the Scope of Accreditation No. 744, which is available at https://www.scc.ca. This accreditation is based on international standards (ISO 17025) and involves extensive site audits and ongoing performance evaluations.

Both laboratories used, BV, and SGS are independent of Centerra.

At BV, samples were prepared as follows (code PRP80-250):

- Samples were dried and weighed
- Entire sample crushed to ≥ 80% passing ~2 mm
- Crushed material was then riffle split and a 250 g sample pulverized to ≥ 85% passing 75 µm.

11.2.3 Assaying

At BV, all drill core samples were analyzed for precious and base metals, as well as multi-elements. Au was assayed using a 30 g fire assay with AAS finish (BV lab code FA430). Gold results over the upper detection limit of the method (≥ 10 ppm) triggered 30 g fire assay with gravimetric finish (FA530). All samples were also analyzed for a 45-element package, including copper and base metals, using a four-acid digestion and inductively coupled plasma mass spectrometry/emission spectroscopy (ICP-MS/ES) on a 0.25 g aliquot (BV lab code MA200). Copper results ≥1% triggered analysis using ICP-MS with AAS finish (MA404) on a 0.50 g aliquot. Silver results ≥100 ppm triggered 30 g fire assay with gravimetric finish (FA530). Sulphur >10% triggered Leco analysis (TC000). Assay methods used by Centerra for Au (FA430, FA530), 45-element package (MA200), and over-limit copper and sulfur (MA404, TC000) are listed on BV's ISO17025:2017 accreditation.

At SGS, samples were analyzed for similar precious and base metals as completed by BV. Gold was assayed using a 30 g fire assay with atomic emission spectrometry (AES) finish (SGS code GE_FAI31V5). Gold over the upper detection limit of the method (≥10 ppm) triggered 30 g fire assay with gravimetric finish (SGS code GO_FAG30V). Samples were also analyzed for a 49-element package, including copper and base metals, using a four-acid digest and (ICP-MS on a 0.2 g aliquot (SGS code GE_ICM40Q12). Copper, lead and zinc were further analyzed by using a four-acid digestion with ICP-AES finish. Sulphur >10% triggered Leco analysis (SGS code GE_CSA06V).

Assay methods used by Centerra for gold, silver and copper are listed under SGS ISO17025:2017 accreditation.

A subset of 2024 samples were analysed for total sulphur and total carbon at BV Labs. One sample in every five samples from the 2024 drill program (total of 1,321 samples) were submitted for Inorganic Carbon by Coulometry Method (TC906 C) and Leco Sulphur (TC000-S). Inorganic carbon is determined by mixing the sample with perchloric acid and the resulting CO_2 is then reacted with monoethanolamine and a pH indicator to form a titratable acid. Color change during the electrolytic titrations is monitored by a photodetector. Leco S involves the addition of Induction flux to the prepared sample, which is then ignited in an induction furnace. A carrier gas sweeps up released sulphur to be measured by adsorption in an infrared spectrometric cell. Results are total and attributed to the presence of sulphur in all forms.

11.2.4 Quality Control

For 2019–2024 diamond drilling, certified reference materials (CRMs), blanks, field duplicates, and coarse lab prep duplicates were used to monitor QA/QC of the core sampling, processing, and assaying processes. While sampling drill core, the logging geologists inserted CRMs and coarse blank samples alternately into the sample sequence every 10 samples. In 2019 programs, CRM samples were inserted into the sample sequence at a rate of 1 in every 20 samples, whereas blank samples were inserted into the sample sequence 1 in every 40 samples. In addition, field and preparation duplicates were inserted alternately into the sample sequence at an approximate rate of 1 duplicate for every 20 samples, with three out of four duplicates being preparation duplicates. Table 11-1 details the quantity of each type of QA/QC sample sent for analysis by year for Centerra drill programs.

Table 11-1: Quantity of QAQC samples submitted for analysis by year

Year	QAQC/Sample type	Qty
2019	Blank	131
	Lab prep-duplicate	166
	Field duplicate	56
	Primary sample	3,874
	CRM	218
2020	Blank	123
	Lab prep-duplicate	183
	Field duplicate	63
	Primary sample	4,292
	CRM	248
2024	Blank	302
	Lab prep-duplicate	303
	Primary sample	6,638
	CRM	301
Grand total		**16,898**

Clean coarse marble landscape rock weighed in ~1 kg samples were used for blank material. One copper-gold CRM was purchased from CDN Resource Laboratories Ltd in Delta, British Columbia, and seven different copper-gold and multi-element certified CRMs from OREAS North America Inc. in

Sudbury, Ontario. Table 11-2 details CRMs used at Kemess between 2019 and 2024. Since 2023, OREAS CRMs have been commercialized in North America by Analytichem, located in Baie-d'Urfé, Quebec. The CRMs were selected to match low, medium and high-grade mineralization ranges and are dominantly sourced from copper-gold bearing porphyry intrusive rocks. The different CRM ranges were cycled through randomly during standard insertion. However, when possible, CRMs were matched to visually estimated grades in mineralized zones. The CRM material submitted was greater than 100 g to ensure reanalyses could be completed if necessary.

Table 11-2: CRMs selected for 2019-2024 Kemess drilling programs

Standard	Au (g/t)	Cu (%)	Ag (ppm)	Mo (ppm)	Grade	Source
CDN-CM-43	0.309	0.233	-	-	Low copper, mid gold	Mt Polley alkalic, marginally silica-undersaturated intrusions, and magmatic hydrothermal breccias. Mineralization occurs in almost all constituent rock types of the MPIC
OREAS 152b	0.134	0.375	0.861	81	Mid copper, low gold	Copper ore drilling reject material from 27 drill holes from the Waisoi district, Viti Levu, Fiji, with the addition of a minor quantity of Cu concentrate (0.6%).
OREAS 502c	0.488	0.783	0.779	226	High copper, mid gold	Blend of porphyry copper-gold ore, barren granodiorite and a minor quantity of Cu-Mo concentrate. The ore was sourced from the Ridgeway underground mine located in the Cadia Valley Operations situated in central western New South Wales, Australia.
OREAS 503c	0.698	0.538	0.83	318	Mid-high copper, high gold	Blend of porphyry copper-gold ore, barren granodiorite and a minor quantity of Cu-Mo concentrate. The ore was sourced from the Ridgeway underground mine located in the Cadia Valley Operations situated in central western New South Wales, Australia.
OREAS 503e	0.709	0.531	1.52	343	Mid-high copper, high gold	A blend of porphyry copper-gold ores, barren granodiorite and a minor quantity of Cu-Mo concentrate. The ores were sourced from both the Cadia Mine and Northparkes Mine. Both mines are located in the central west of New South Wales, Australia.
OREAS 504d	1.46	1.10	2.69	507	High copper, high gold	A blend of porphyry copper-gold ores, barren granodiorite and a minor quantity of Cu-Mo concentrate. The ores were sourced from both the Cadia Mine and Northparkes Mine. Both mines are located in the central west of New South Wales, Australia.
OREAS 507	0.176	0.622	1.34	114	Mid-high copper, low gold	A blend of porphyry copper-gold ore, barren granodiorite and Cu-Mo concentrate. The ore was sourced from the Northparkes Mine located in the Central West of New South Wales, Australia.
OREAS 609b	4.97	0.498	24.6	5.54	Mid-high copper, very high gold	A blend of gold-copper-silver bearing ores from Evolution Mining's Mount Carlton Operation in Queensland, Australia and barren rhyodacite rock sourced from a quarry east of Melbourne, Australia.

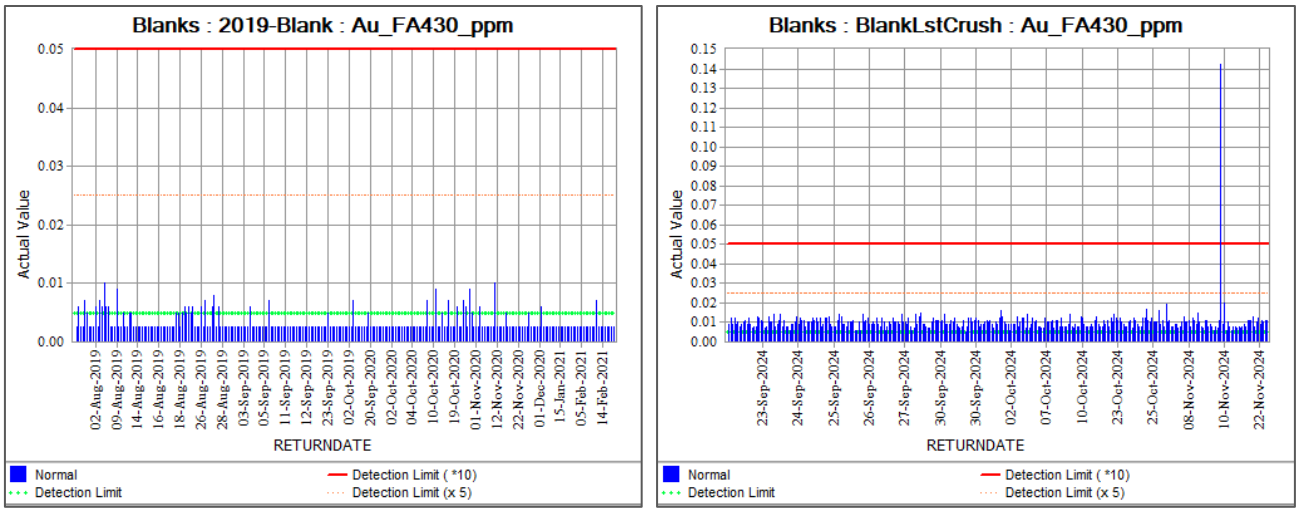
In addition to the inserted reference materials, field and coarse reject duplicates were inserted alternately into the sample sequence every 20 samples. Field duplicates were prepared by quartering one half of the core, with one quarter sent for analysis with a unique sample ID, and the other remaining in the core box. Coarse reject duplicates were prepared at BV labs prior to sample pulverization by

taking a second 250 g riffle-split. The QA/QC insertion rates are acceptable according to current CIM best practice standards, with QA/QC samples accounting for ~15% of the 2019–2024 assay database.

After completion of the program approximately 3.5% (2% between 2019–2020 and 5% in 2024) of the sample pulps from BV were submitted to the SGS laboratory in Burnaby, British Columbia as an independent check for analytical bias and accuracy.

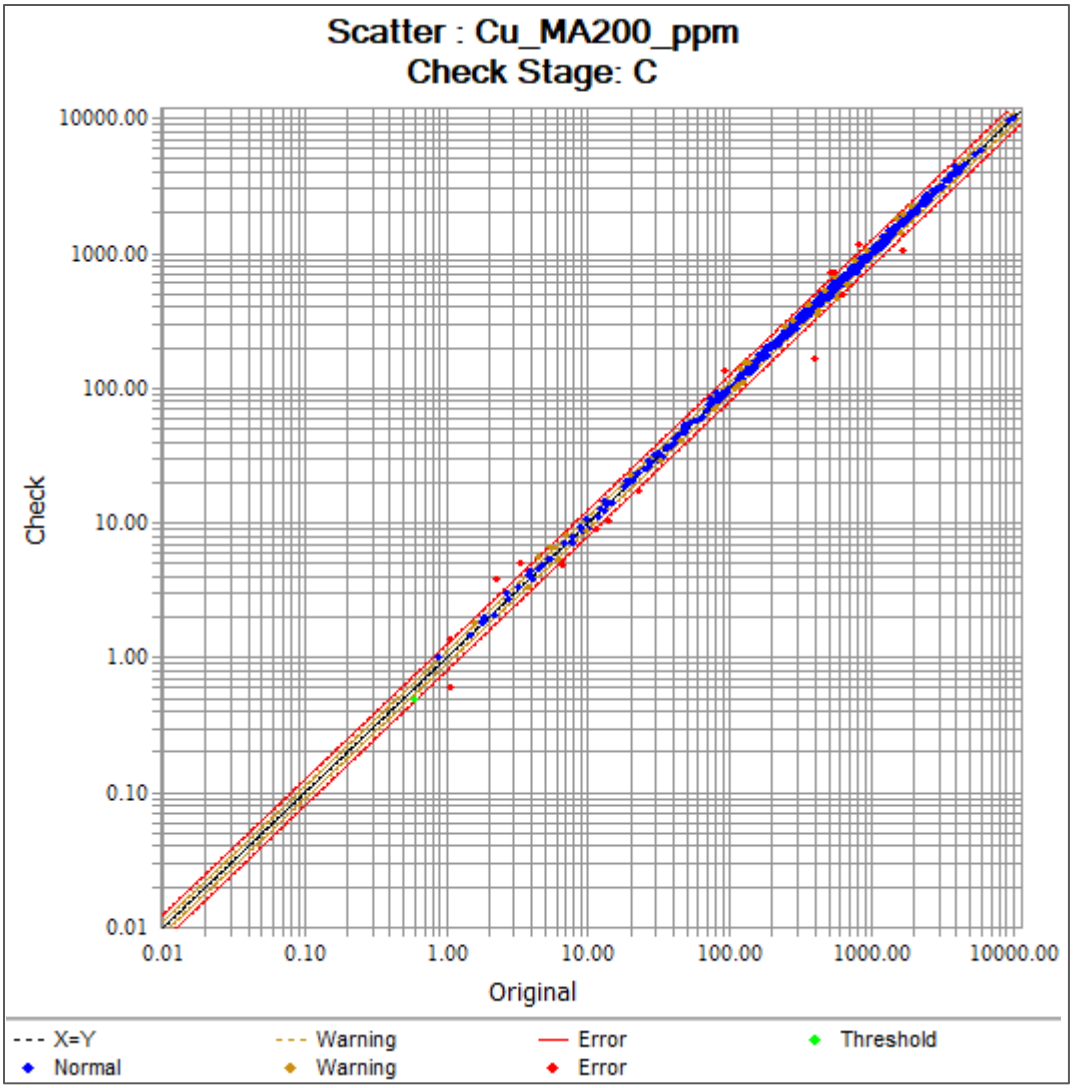
Evaluation of gold and copper analyses of quality control blanks indicates that no significant or systematic contamination or laboratory error occurred during the course of the 2019–2024 programs (Figure 11-1).

Figure 11-1: Blanks for gold fire assay (FA430) from 2019–2024



Lab preparation and analytical precision were examined using matched pairs created by comparing coarse reject duplicates to the original samples. Scatter plots of matched-pair duplicates for both gold and copper correlate well, demonstrating the data is precise. For example, for copper duplicates from 2019-2024, with a samples size of 673, the Coefficient of Determination (R^2) = 0.9985, and only 3.8633% of the sample population with a relative difference >15% between duplicate and original crush sample, well below the industry standard of 10% of the population with >20% relative difference (Figure 11-2).

Figure 11-2: Coarse reject duplicates for copper (MA200) from 2017–2025



After the assay results were received from the lab, gold and copper assays were checked by a Centerra database manager using control charts for the CRMs, blanks and duplicates. The procedure for quality control failures (samples bracketing CRMs with assay values ± three standard deviations of the expected value) included documentation and relevant batches of samples were requested for re-assay by BV labs using the primary pulp. If failed standards performed acceptably on the second run, then the original assays were corrected and new certificates were issued for the batches of associated samples. Standard failures reproduced on the second run were deemed to be due to normal variation in the CRM and therefore the original results were accepted as accurate. Overall, the standards performed well and are considered acceptable, with the bulk of the data plotting within the ± 3 STD range. A representative QA/QC plot for both copper and gold standard OREAS 152b is shown in Figure 11-3 and Figure 11-4.

Figure 11-3: Standard OREAS 152b by sequence for copper (MA200)

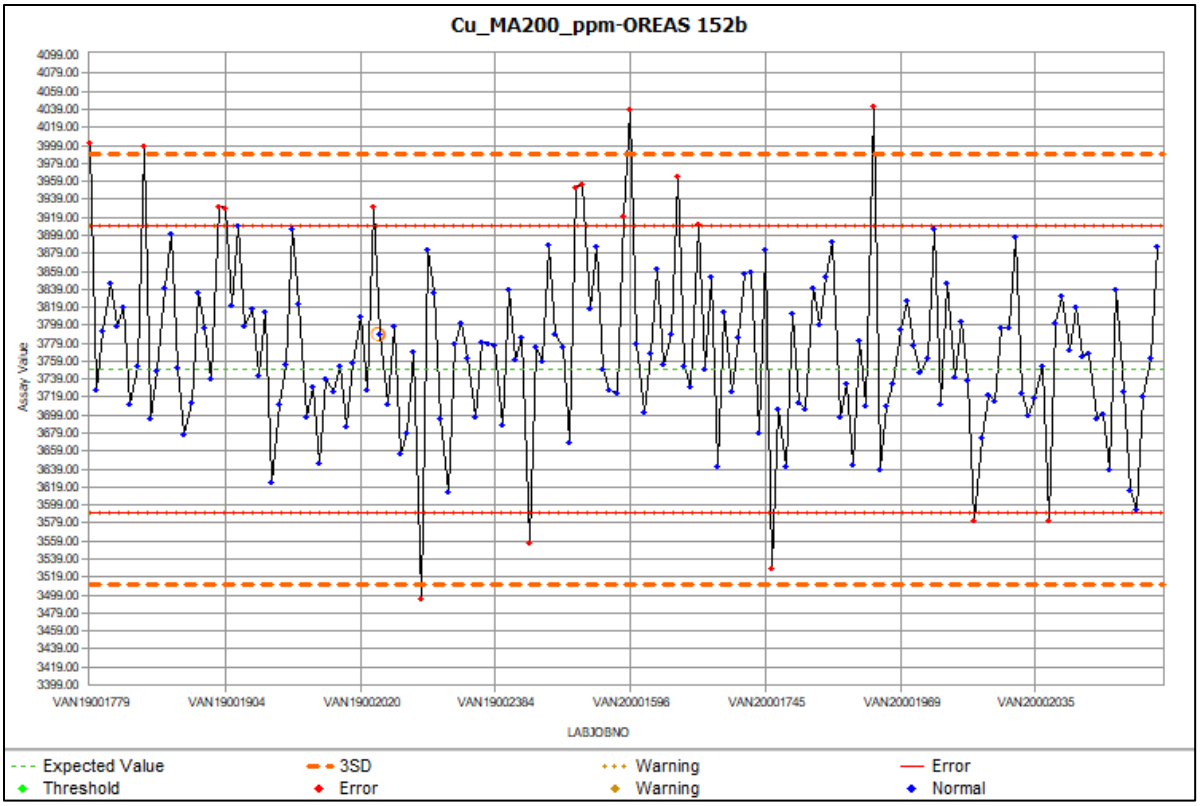
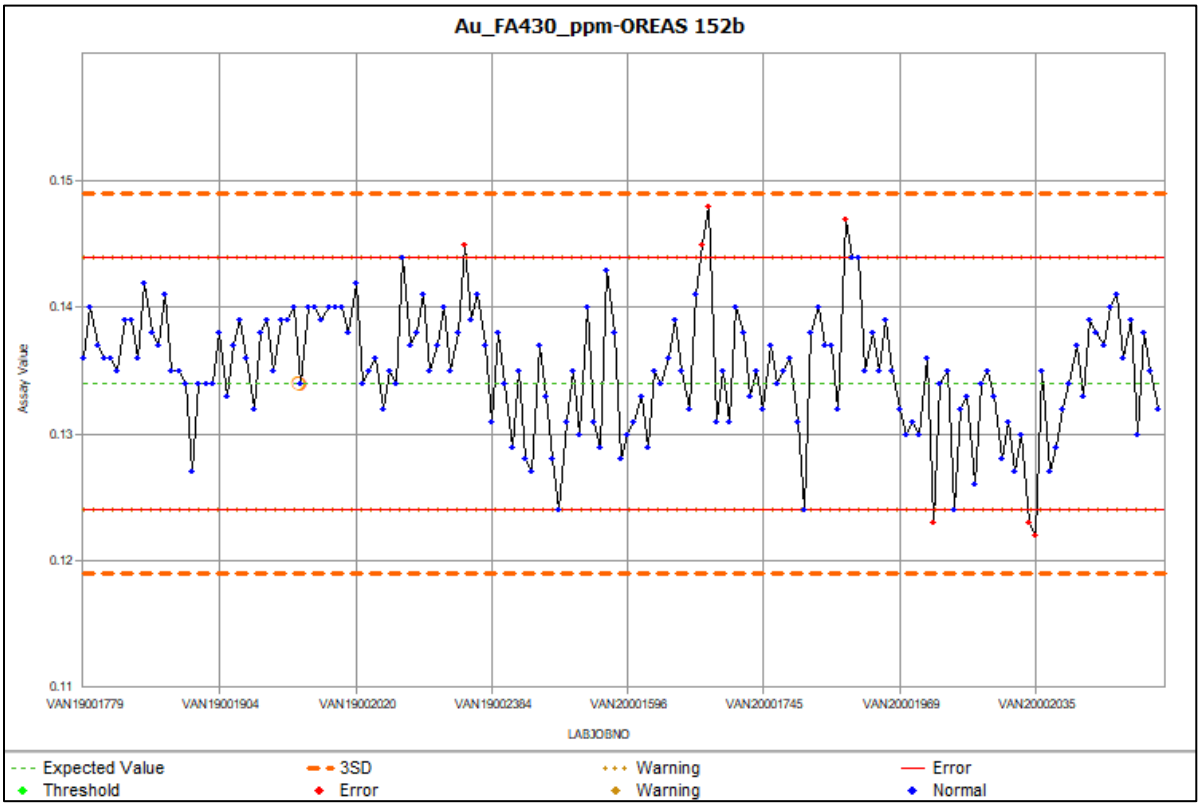


Figure 11-4: Standard OREAS 520 by sequence for gold (MA200)



The QA/QC analysis of drill core from 2019 to 2024 demonstrates that assay data are free of contamination and adhere to industry standards and in the QP's opinion, of sufficient quality to be used in a resource estimation.

11.2.5 Security

After drill hole samples were cut and bagged in poly-plastic sample bags sealed with a plastic zip-tie, then were prepared for sample shipment by a geotechnician or core-cutter. Individual sample bags were collected in labelled rice bags and sealed with zip ties and individually numbered security tags. A copy of the sample submission form with a list of samples and security tags for each batch was included in the last rice bag of samples and emailed to the exploration office and BV labs. Samples were shipped via Kemess warehouse services and Bandstra Transportation Systems Ltd to Bureau Veritas Canada Inc. ("Bureau Veritas") laboratories in Vancouver.

All Kemess drill core is stored at the Kemess mine site. Once the core had been sampled, aluminum tags were affixed to the boxes with the Hole ID, depth, and box number handwritten on the tags. The boxes were then transported to the core farm area (637614E, 6321312N) at Upper Camp where historical Kemess core is stored. Core was stacked in sequential order on racks made from 4"x4"X16' timbers.

11.2.6 Adequacy of Sample Preparation, Analysis and Security

In the opinion of the QP for this Item of the Technical Report, sample preparation, security, and analytical procedures utilized during drilling programs were adequate and conducted according to CIM Estimation of Mineral Resources and Mineral Reserves best practice guidelines.

11.3 PRE-2019 SAMPLES – KEMESS SOUTH

11.3.1 Sampling Method and Approach

Various sampling protocols were used for the work at Kemess South over the different drill campaigns and operators. Sample length was determined by the geology of the deposit; sample lengths were generally 2 m in length and respected lithologic boundaries. In more recent work, starting with Northgate in the 2000s, geologists sampled the full length of core drilled.

Work completed by employees of the company included core logging, sample layout, sample splitting and preliminary sample preparation. A professional geologist oversaw all the work from core logging to sample splitting and preliminary sample preparation, and shipping.

11.3.2 Sample Preparation and Laboratory

The drill core was split using a rock saw, and then samples were passed through primary crushing. During the 2002 program, a portable sample preparation lab was leased from ALS Chemex. For the 2003 to 2006 programs, a sample-bucking facility was built near the mill area. The core samples were dried at 105°C for 3 hours then crushed to 80% passing 10 mesh using a Rhino jaw crusher at the mine site. Each sample is riffled twice to approximately 250 g using a Jones Riffle with one split being retained at the mine, and a 250 g sample sent to the lab. The remainder of the sample was discarded. The portion of sample retained at the mine site is kept in a plastic bag with a sample tag and stored in a plastic pail (Skrecky 2007).

11.3.3 Assaying

All the samples from the drilling programs since 1999 have been analyzed off-site at commercial laboratories. The 1999 series of holes and a few of the 2006 samples were analyzed at the Kemess assay lab in 2000 and 2006. The assay lab has an industry standard quality control and assurance program. The assay values for the 1999 drilling were checked against the blasthole values in the drillhole vicinity.

Copper and gold assays for the 2000 and 2006 programs were done by Assayers Canada Laboratory in Vancouver and for the 2001 program by Bondar-Clegg Laboratory in North Vancouver. All samples are assayed for gold by fire assay and atomic absorption techniques and for copper by atomic absorption. The 250 g samples are pulverized to 90% passing minus 150 mesh to make a sample pulp. Copper assay was done by triple acid digestion, HCl - HNO_3 - HBr, 2 g, digestion in Teflon beakers, with an atomic absorption finish. Samples were also submitted to a one assay-tonne gold fire assay, 30 g nominal sample weight fire assay fusion by lead flux with Ag collector, with an atomic absorption finish. A separate multielement scan was carried out using an aqua regia acid digestion and ICP-AES analysis. All sample batches were subjected to Assayers Canada's internal quality control procedure.

Chemex Labs is widely used by the mining and exploration industry and carries the highest certification as registered assayers, including ISO 9002, ISO: 9001:2000, and they are working towards ISO 17025. Assayers Canada, another widely used lab with ISO: 9001:2000, completed most of the assaying for the 2006 program with a minor number of samples also processed at the mine site assay lab.

11.3.4 Quality Control

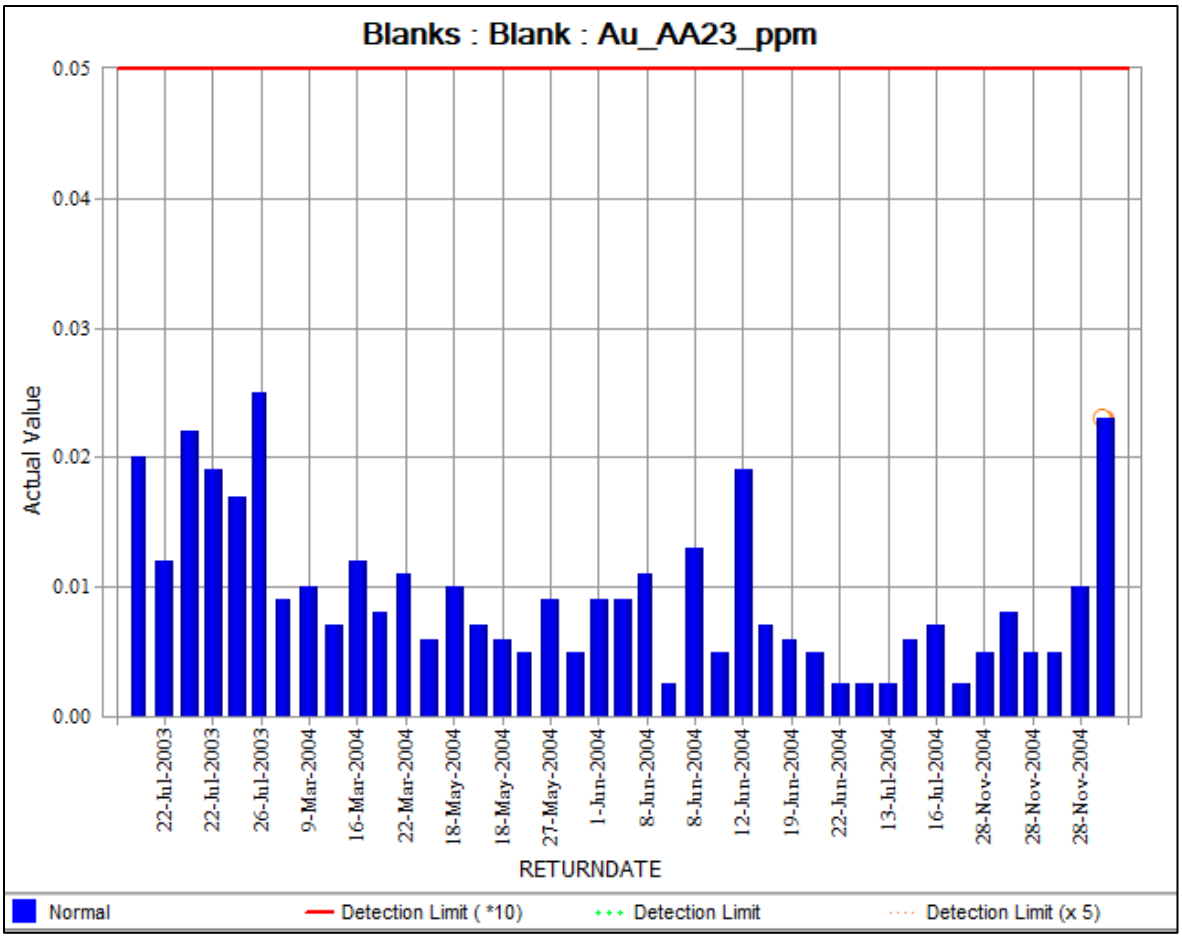
The 2007 Kemess South technical report by Skrecky describes the historical quality control procedure which included the routine introduction of blanks, duplicates and standards, routine verification of primary crushing and pulverization through screen analysis. Commercially prepared certified standards from Rocklabs were used in the QAQC. These standards were selected to closely match the matrix of

the material sampled and appropriate grade ranges. Reject duplicates were used to measure the precision of the lab used.

In 2025, Centerra reviewed the Kemess South drilling QA/QC procedures and did not find records of QA/QC samples prior to 2003. The QA/QC data available was for 2003–2005 drill programs and is reviewed below. The QP recommends re-assaying a selection of pre-2003 historical drill core if available to verify the historical assay results. If the historical drill core is unavailable, the QP recommends drilling a selection of twin holes (e.g. 10% of the total samples) for assay and resource model verification.

The data for blank samples show no significant or systematic contamination or laboratory error occurred during the course of the 2003–2005 programs (Figure 11-5).

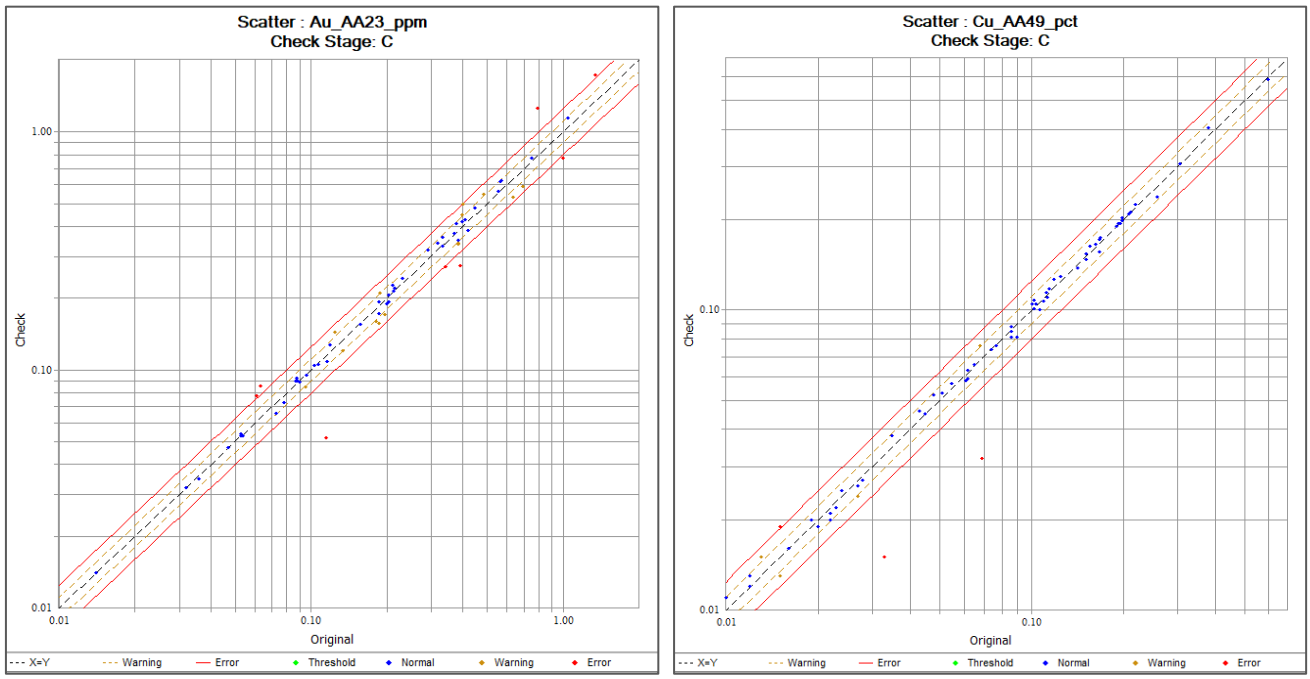
Figure 11-5: Blanks for gold fire assay (AA23) from 2003–2005



Lab preparation and analytical precision were assessed using matched pairs created by comparing coarse reject duplicates to the original samples. Scatter plots of matched-pair duplicates for both gold and copper show strong correlation, indicating high precision in the data. For example, for gold and copper duplicates from 2003–2005 (sample size: 73), the Coefficient of Determination (R^2) values are 0.9664 for gold and 0.9974 for copper. Only 16.44% of the gold population and 6.85% of the copper

population exhibit a relative difference greater than 20% between duplicate and original samples (Figure 11-6). These values are below the industry standards of 20% for gold and 10% for copper, respectively.

Figure 11-6: Coarse reject duplicates for gold (AA23 – ppm) and copper (AA49 – %) from 2003–2005



The reference materials used during the 2003–2005 programs were certified only for gold. Overall, the standards performed well and are considered acceptable. Representative QA/QC plots show expected values at 0.651 ppm and 1.298 ppm of gold for standards OxE-21 and OxH-29 (Figure 11-7, Figure 11-8).

Figure 11-7: Standard OxE-21 by sequence for gold (AA23)

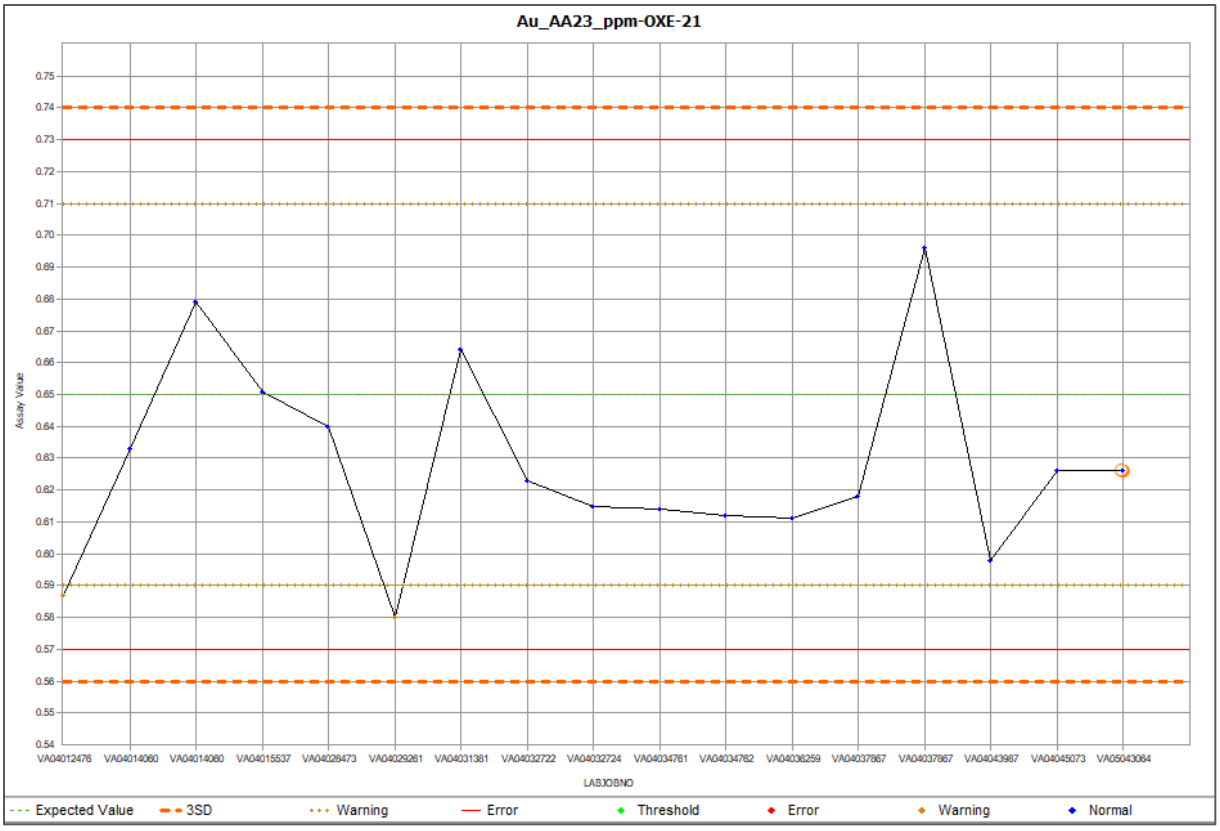
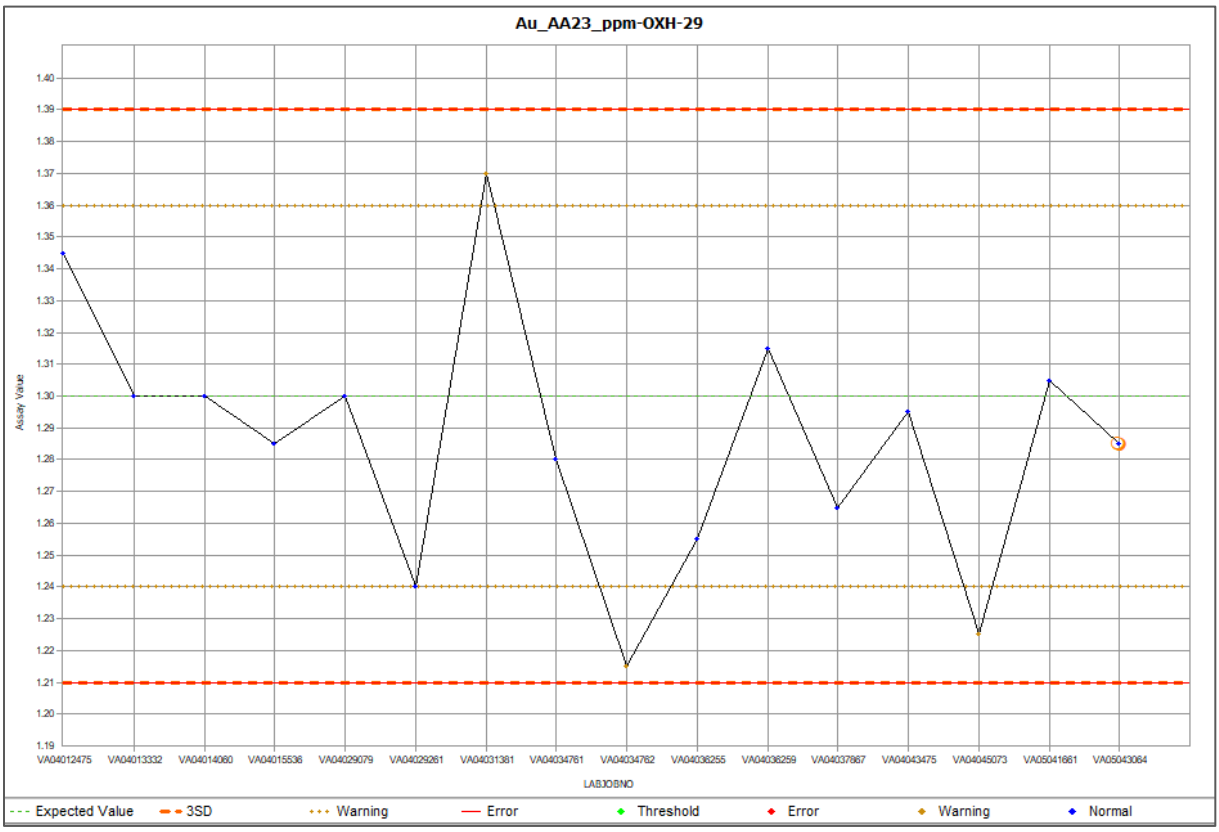


Figure 11-8: Standard OxE-29 by sequence for gold (AA23)



11.3.5 Security

The 2007 Kemess South technical report by Skrecky describes the sample security measures for historical drill samples. The portion of the sample sent to the lab was placed in a plastic bag with a sample tag, shipped in a plastic pail with two security tags, the pail top was sealed and taped, by bonded air courier to an independent commercial lab in Vancouver for gold and copper analyses. A submission sheet was sent along with each pail of samples that included the name of the sample preparation person, the date, the sample numbers, the number of samples, and the numbers of the security tags.

The Kemess lab performed numerical control checks when the drill core samples are received for sample preparation and sample pulp packing. All coarse rejects are stored inside at the mine site. The core storage site near Kemess Lake is a well-organized facility. The remaining cores are still in core boxes and are available for geology reviews as well as check assays.

11.3.6 Production Blast Hole Samples during Operations

Historical blast hole sample results at Kemess South were not verified and not used for Mineral Resource estimates.

11.3.7 Adequacy of Sample Preparation, Analysis and Security

In the opinion of the QP for this Item of the Technical Report, sample preparation, security, and analytical procedures utilized during Kemess Main Zone drilling programs were adequate and conducted according to CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The QP recommends re-assaying a selection of pre-2003 Kemess South historical drill core, if available, to verify the historical assay results. If the historical drill core is unavailable, the QP recommends drilling a selection of twin holes (for example 10% of the total samples) for assay and resource model verification.

12 DATA VERIFICATION

12.1 GEOLOGICAL DATA

Kemess has established and documented procedures for verifying and validating exploration data. Experienced Centerra geologists and staff implement industry standard practices to ensure a high level of confidence in exploration data. All exploration data are verified and validated prior to being considered for geological modelling and Mineral Resource estimation.

Centerra technical staff monitor quality control data on a continual basis. An acQuire database is used to manage exploration data. The database management system includes tools for quality control and ongoing monitoring and reporting. Investigating, and taking appropriate actions of quality control failures are part of the data verification process, which may include re-assaying of samples.

12.1.1 Site Visits

The QP for mineral resources and reserves has visited the site July 8–10, 2025; the site visits included reviews of core logging facilities, open pit mine, TSF, processing plant and maintenance facilities. The QP reviewed core logging procedures and found them to be adequate for accurately representing the lithologies, alteration types and rock mass characteristics of the deposit.

12.1.2 Database Verification

Coordinate System

In 2024, the Centerra database management team, validated and imported historical data to the Kemess acQuire database. In this database review, several unclear items were addressed including the historical Kemess Mine Grid. All historical Kemess engineering data has been designed in the local ground coordinate system (e.g. block models, Kemess South pit, Kemess South infrastructure, KUG infrastructure).

A difference of ~1.87° shift in between the Kemess Mine Grid (aligned with True North) and NAD83 UTM Zone 9 at the Kemess property. When using the Kemess Mine Grid X,Y,Z coordinates the Azimuth_Kemess_MG value should be used in the Survey table.

Multiple different transforms between NAD83 UTM Z09N and the mine grid have been created. The drilling database has utilized the following 2001 transform for consistency despite newer ones surveyed in 2007 and 2017 based around the Kemess South pit and tailings storage facility.

To convert from Mine Grid to UTMs:

- Scale: 0.9997655 Rotate: -1.872777 Shift: N +6309986.944 E +626479.931

To convert UTMs to Mine Grid:

- Scale: 1/0.9997655 Rotate: +1.872777 Shift: N -6309986.944 E -626479.931

For the Centerra geological drilling and surface sampling, a two-point transform was created from the above transformation using the two points detailed in Table 10-3.

A detailed description of data collection for the historical core drilling programs pre-2016 are described in the previously published Kemess North Technical Report by Golder Associates (2017). For details on Centerra-AuRico drill programs from 2017–2025, refer to Item 10 of this report.

12.1.3 Assay Quality Control

Details of the historical drill programs including sampling method and approach, sample preparation and laboratory, assaying, quality control, and security, have varied with the different programs and operators throughout the project's history. A detailed description of assay control programs for the historical diamond drilling programs (pre-2016) is described in the previously published Kemess North Technical Report by Golder Associates (2017). Centerra-AuRico programs (from 2017–2025) are included in Item 11 – Sample Preparation, Analyses, and Security of this report.

2017 to 2025

QA/QC procedures implemented for the 2017–2025 drilling programs are described under the Quality Control subsections of Section 11.

Further to the QA/QC procedures described above, routine data checks are performed to ensure the assays in the drill hole database are checked against assay certificates received by the lab.

In 2025, the Centerra database management team developed a series of structured SQL queries to assess the integrity, completeness, and consistency of assay data within the acQuire database. These queries target key validation points such as interval logic, overlapping and duplicate samples, NULL values, unrealistic assay values, and QA/QC compliance. The logic is tailored to the custom field naming conventions used in the current assay table (sampfrom, sampto, and sampleid) and supports a systematic audit of legacy and current datasets. Each query contributes to identifying potential data quality issues that could impact downstream geological modelling and resource estimation processes.

As part of the audit process conducted directly on the acQuire database, a set of QA/QC reports was prepared using the acQuire QA/QC Report object to evaluate the analytical reliability of the assay data. These reports represent a standalone quality assessment, independent of the export mechanisms, and are intended to provide additional confidence in the integrity of the underlying data.

Given that multiple laboratories have been engaged over the years, individual QA/QC reports have been compiled and presented separately for each laboratory.

12.1.4 Drill Hole Database

The structure of the acQuire database follows the standard acQuire GIM Suite data model, which is a structured and relational schema specifically designed for managing geoscientific data. The model ensures consistent handling of drillhole-related information such as collar locations, downhole surveys, geological logs, and assay results, with built-in referential integrity across interconnected tables.

A significant portion of the data currently housed in the database was imported as legacy data prior to 2019. These legacy datasets were migrated from various historical sources by the previous database administrator during the initial implementation phase. Since 2019, the Kemess Exploration Team has been actively entering new data and performing data imports in accordance with standardized data entry procedures and acQuire workflows.

The acQuire database is hosted on a SQL Server instance managed by the Global Exploration Information Management and corporate IT team. The database is accessed through the acQuire GIM Suite and is linked to a centralized file repository for related workspace configurations. Users must have appropriate read/write permissions assigned by IT and access rights configured in the acQuire user group settings. Access credentials and permissions are managed by the database administrator in coordination with IT security protocols. Any issues related to connectivity or access rights should be directed to the designated data manager or the IT helpdesk.

12.1.5 2025 Centerra Database Audit

Throughout 2024–2025, additional validations and verifications of the database were conducted during the migration to acQuire data systems management software. The last audit Kemess Exploration Database Internal Audit, April 2025. This audit aims to evaluate the integrity, consistency, and completeness of geological drillhole data stored in the acQuire database. The review focuses on identifying potential data quality risks or gaps resulting from both historical legacy data imports and ongoing real-time data entry processes. As part of this audit, the ReadMe documentation was also cross-checked to validate the structure and content of legacy records and to clarify any historical data entry conventions.

The audit scope and methodology included:

- Legacy data imports – A significant portion of the database consists of historical drillhole data imported during the initial implementation phase. This data was migrated from various sources, and its validation relied heavily on the availability of supporting ReadMe files, metadata, and source documentation. For this dataset, the audit focused on: Verifying the

completeness and integrity of imported records. Cross-checking legacy coding and field conventions. Identifying any inconsistencies introduced during bulk imports. Confirming that legacy data aligns with the current acQuire schema structure.

- Recent data (2019–present) – Since 2019, the Kemess Exploration team has actively been entering and importing new drilling data directly through acQuire, using established workflows and validation protocols. For this data stream, the audit included listing the application of acQuire's standard data entry workflows.

Collar validation included primary key enforcement, mandatory fields check (e.g. HOLEID) relational integrity and coordinate range rules.

Downhole survey validation included parent record requirement for relational integrity, depth sequence validation, mandatory field checks, azimuth/dip value ranges, and duplicate entry control.

Downhole survey data was audited to identify potential errors in drillhole orientation, which can significantly impact 3D modelling and interpretation. The review focused on validating azimuth and dip values, checking for missing or duplicated entries, and confirming that all records fall within acceptable numeric ranges and are consistent with the collar reference system.

In acQuire, location data is ranked based on predefined spatial accuracy tiers, allowing for the differentiation between precise survey data (e.g. differential GPS) and less reliable inputs (e.g. handheld GPS or estimated positions). During this audit, the spatial reference system definitions, coordinate integrity, and ranking classifications of collar data were evaluated to ensure alignment with corporate standards and industry best practices.

Additionally, in the field definitions, the prefix **BEST_** is used to indicate the best available location record rank for a given collar, particularly when multiple location sources (e.g. original GPS reading, re-surveyed coordinates, or adjusted positions) exist. This allows the system to prioritize and apply the most accurate spatial reference during data export and modelling workflows.

12.1.6 Data Source of Resource Model (acQuire and Leapfrog Software)

Throughout 2024–2025, project-specific database accessed via ODBC connection to the acQuire database, along with their form definitions and any filters applied at the form definition level, if applicable. Resource estimation studies are conducted using Leapfrog Geo and the Edge module. Project versioning and tracking are managed through Seequent Central Portal.

The Export Profile in acQuire is a predefined configuration that defines which tables, fields, and filters are included in the data export. It ensures consistency by specifying output formats (e.g. CSV, Excel), column headers, and data structure, allowing for smooth integration with Leapfrog Geo. This integration

streamlines the data pipeline from collection to interpretation, enhancing both efficiency and confidence in geological modelling processes.

Key settings and checks are applied through the **Export Profile** for the extraction of data from the acQuire database. This includes configurations related to the **Collar**, **Survey**, **Geology**, and **Assay** datasets, ensuring that the appropriate and validated information is retrieved for further processing and analysis.

The reporting and audit activities conducted provide a comprehensive validation of the data workflows established within the acQuire database. Each stage – ranging from the evaluation of Collar, Survey, Geology, and Assay data to the review of Export Profiles and integration pathways – demonstrates that the current system supports a structured and traceable flow of geological information. The consistency and readiness of data for downstream use, including live connections with modelling platforms, are further evidence of the robustness of these workflows.

To ensure continuity and integrity in data management, all data exports should be performed in coordination with the database administrator. Following export, it is recommended that responsible technical users apply independent validation workflows. This reinforces shared accountability and supports consistent data quality across all operational stages.

The data reviews found the assay data acceptable, and the QP considers the final 2025 drill hole database to be robust and verified.

12.1.7 Independent Logging and Sample Verification

After completion of the Centerra drill programs (from 2019–2024), approximately 5% of the sample pulps from BV were submitted to the SGS laboratory in Burnaby, British Columbia as an independent check for analytical bias and accuracy.

12.2 GEOTECHNICAL DATA

The project area has been the subject of multiple technical studies and development initiatives for both open pit and underground mining methods. A significant amount of technical information is available. A review and gap analysis of previous geotechnical studies and site investigations were performed for the Main Zone and Kemess South open pit area. The existing geotechnical dataset (2001–2011 drillholes) is adequate for this stage of study. Good correlation with logging data was observed especially within the 2010 drilling program.

A separate review and gap analysis for KUG deposit was performed and focused on the evaluation of the proposed long hole open stoping mining method and identifying areas of further work required for future stages.

The QP has determined that the geotechnical data is adequate to support the MRE, the economic analysis and the conclusions contained in this technical report.

12.2.1 Geomechanical Logging QAQC

All available drilling data in proximity to the underground workings was reviewed. The review included:

- Comparison of core photos to the logged rock mass qualities for the SRK drillholes.
- Core photos are not available for the AMC drillholes. As a result, the data were reviewed for consistency (i.e. correlation between associated parameters such as RQD and fracture frequency (FF)/m) for the AMC and 2011 SRK drillholes.
- Comparison of the SRK and AMC rock mass quality logs with Knight Piésold Ltd (KP) drillholes close by where possible.

The data review showed that the 2003 KP, 2011 SRK, 2020 SRK, and 2024 KP data were generally consistent and aligned with the current geotechnical interpretations.

12.2.2 Geomechanical Orientation Data QAQC

The SRK and AMC oriented core data around the underground workings was reviewed. Drillhole data was plotted to assess if the orientations of discontinuities aligned with the orientations for the KUG area. The SRK data showed a strong northeast-dipping joint set. However, the data is limited and the drillholes are biased due to their orientation. The AMC data shows east to north dipping joint sets. However, 21 out of the 26 oriented drillholes having roughly the same orientation resulting in a significant bias in the results.

12.2.3 Review of Laboratory Data

The lab data density and distribution are suitable for this level study, but additional samples should be collected to validate the historic lab testing results.

12.3 METALLURGICAL DATA

Metallurgical testwork supporting this Technical Report includes samples collected from the Main Zone open pit in 2025, as well as historical samples collected from the underground drilling programs of 2011, 2012, 2014, 2015, and 2019. All samples used for metallurgical testing were prepared from portions of geological samples collected from site, as described in the preceding sections of this Technical Report. Additional details regarding the metallurgical test programs and results are provided in Item 13.

The QP has reviewed the available metallurgical test results and associated documentation relevant to the development of the metallurgical recovery model. This review included an assessment of the test methodologies, results, and underlying assumptions used in estimating metallurgical recoveries for use

in the overall financial model. In the opinion of the QP, the metallurgical data and assumptions are appropriate and sufficiently reliable for the purposes of mineral resource evaluation and economic analysis presented herein.

12.4 ENVIRONMENTAL DATA

Environmental data collected at the Kemess site to support ongoing care and maintenance, technical modelling, baseline environmental studies, and effects assessments were obtained using procedures consistent with the BC Field Sampling Manual (BC ENV 2013). All samples were submitted to an accredited laboratory, and quality assurance and quality control (QA/QC) samples represented a minimum of 10% of the total dataset, including field blanks, equipment blanks, trip blanks, and duplicate samples. QA/QC results were evaluated against established data quality objectives and departures from objectives were interpreted accordingly. Field instruments were calibrated and verified before use.

Environmental data, QA/QC records, and instrument calibration records are retained as part of the environmental monitoring program for the site. Kemess remains in compliance with the data verification requirements of all active environmental permits. The sampling methodologies, analytical results, and QA/QC performance were reviewed by a QP and deemed adequate and reliable for the studies described in this report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

This section summarizes the mineral processing and metallurgical testwork supporting the PEA. The results establish the comminution design parameters, flotation performance expectations, and metallurgical recovery assumptions applied in the study.

Testwork has been completed on material from both the KUG deposit and the Main Zone. While the Main Zone represents most of the planned mill feed, most historical metallurgical investigations were conducted on KUG material. To support interpretation of the available data, this section is divided into three parts:

- Historical KUG testwork
- The 2025 test program focused on Main Zone material
- A summary of relevant historical operating data.

13.2 TESTWORK OVERVIEW

Multiple metallurgical tests have been completed for the Project, with the majority of historical programs focused around KUG. These programs include mineralogical assessments, comminution characterization, and flotation testing which has provided the primary basis for understanding ore hardness, flotation response, and expected concentrate quality.

The 2025 test program provided additional mineralogy, comminution, and flotation data specifically for the Main Zone material, which forms most of the planned mill feed.

A summary of the historical and recent test programs is provided in Table 13-1.

Table 13-1: Historical testwork summary

Year	Laboratory/Location	Material tested	Testwork focus	Report name
2011	G&T Metallurgical Service	Kemess Underground	Comminution	Metallurgical assessment of five composite samples from the Kemess Underground project KM2911
2012				Ore hardness testing for the Kemess Underground project KM3442
2014	ALS Metallurgy	Kemess Underground	Mineralogy and flotation	Metallurgical testing for the Kemess Underground project KM4379
2015	ALS Metallurgy	Kemess Underground	Comminution and flotation	Metallurgical testing for the Kemess Underground project KM4730
			Comminution	Kemess Underground grindability testing KM4864

Year	Laboratory/Location	Material tested	Testwork focus	Report name
2019	SGS Minerals Services	Kemess Underground	Mineralogy	Kemess Mineralogy 17102-01
			Comminution	Kemess Grindability 17102-03
			Comminution and flotation	Kemess Grindability 17102-04
	Outotec Research Center	Kemess Underground	Comminution	HIGmill ® Testwork Report – HIG5, Report Reference No: 19123-ORC-T
2025	ALS Metallurgy	Main Zone Open Pit	Mineralogy, comminution and flotation	Metallurgical Testing on Life of Mine Samples from the Kemess Project KM7413

13.2.1 2011-2019 Test Program (Kemess Underground)

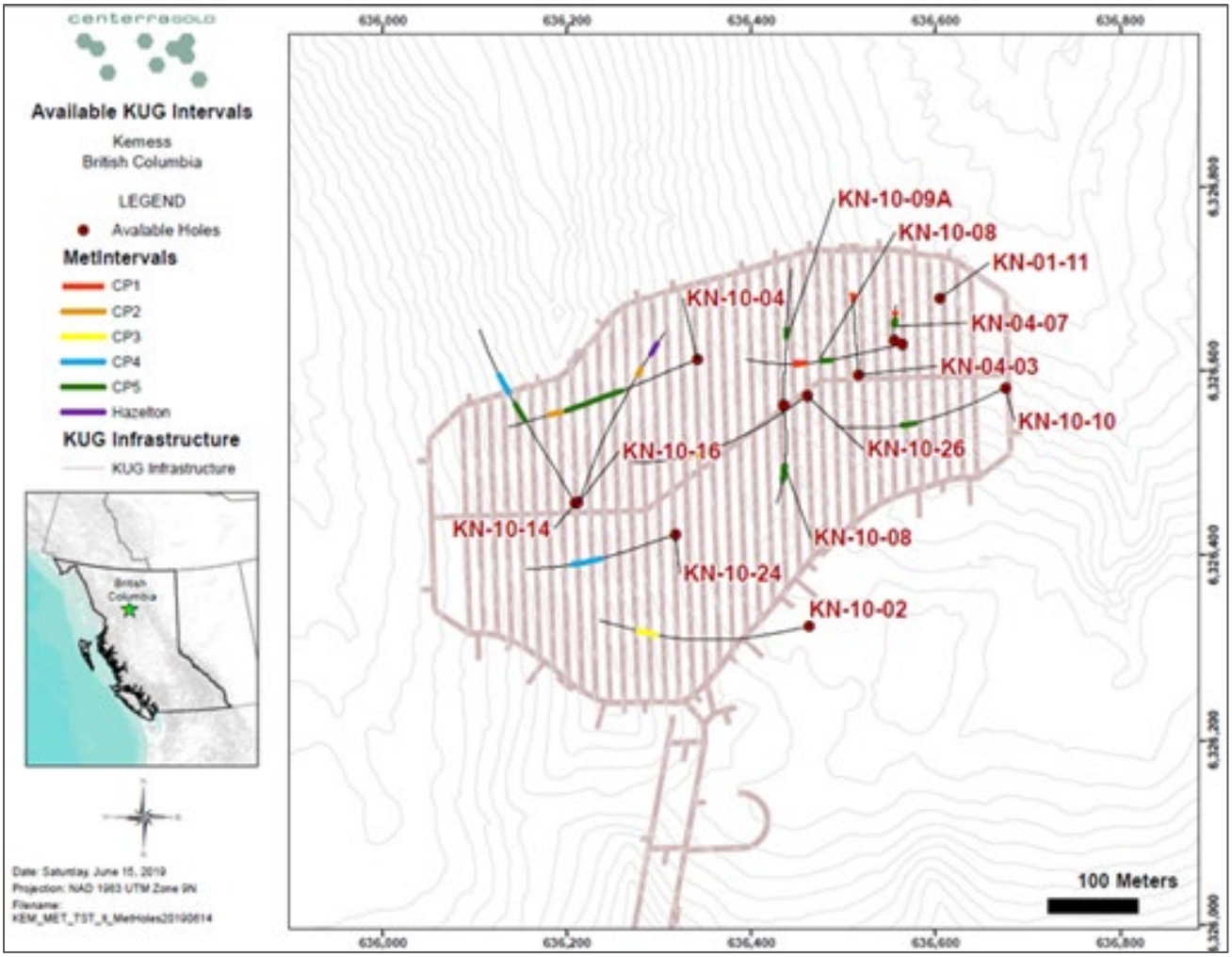
Sample Selection

Five major mineral types and two waste domains were previously defined in the KUG. All samples submitted for the comminution and flotation testwork were selected based on the domain definition shown in Table 13-2. The sample locations in the underground cave are presented in Figure 13-1 and Figure 13-2. Test results summary discussed in the report will only focus on the mill feed material.

Table 13-2: KUG rock domains

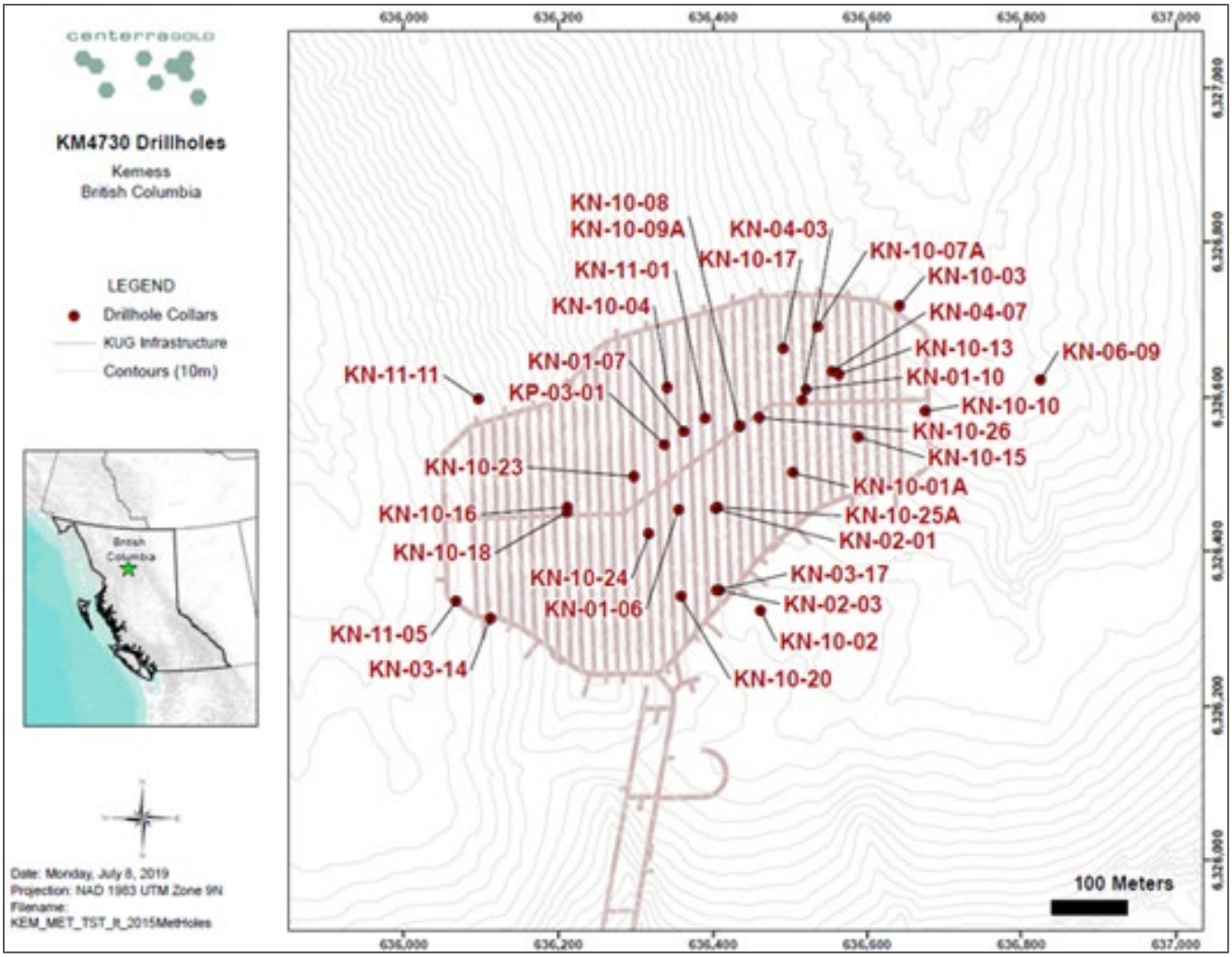
Domain	Rock type
CP-1	HG monzonite
CP-2	MG monzonite
CP-3	LG monzonite
CP-4	Mixed lithology
CP-5	Upper volcanics
CP-6	Dykes (waste)
CP-7	Hazelton (waste)
Others	Development ore

Figure 13-1: Sample locations for comminution testwork (KM3442, 2012)



Note: Footprint is obsolete block cave layout.

Figure 13-2: Sample locations for comminution testwork (KM4730, 2015)



Note: Footprint is obsolete block cave layout.

Head Assays

The Underground domain head samples were submitted for screen metallics, chemical assay analysis, and an inductively coupled plasma (ICP) multi-element scan to characterize the copper, gold, sulphide, arsenic and other elemental content. Results are summarized in Table 13-3.

Table 13-3: Head assay summary (Kemess Underground)

Element	Sample ID				
Unit	CP-1	CP-2	CP-3	CP-4	CP-5
Au g/t (SM)	1.52	0.75	0.51	0.41	1.72
Cu %	0.51	0.35	0.26	0.24	0.23
As %	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001
C(t) %	0.04	0.09	0.12	0.16	0.09
C(g) %	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
TOC leco %	<0.05	<0.05	<0.05	0.09	<0.05
CO_3 %	0.2	0.52	0.59	0.86	0.45
S %	6.22	2.93	2.22	2.25	7.41
$S^=$ %	5.36	2.64	1.77	1.75	3.28
SO_4 %	1.8	0.90	0.80	1.2	11
S° %	<0.05	<0.05	<0.05	<0.05	<0.05

Mineralogy

The KUG deposit is separated into five domains (CP-1 through CP-5) with different lithologies. The mineralogy for each domain was reviewed in KM4379 and 17102-01 and is summarized in the following sections.

Copper Mineralogy

The liberation of copper was reviewed in KM4379 and is presented in Figure 13-3. The sulphide mineralogy of the ore for the different domains is presented in Figure 13-4. The size fraction distribution of the copper is presented in Table 4-1 and was taken from the updated mineralogy report from Project 17102-01 (SGS, April 25, 2019). The data from these two programs is summarized below:

- Copper is present primarily in the form of chalcopyrite. There are negligible amounts of secondary copper minerals for all five domains. There are also negligible amounts of copper oxides for all domains reported in both the KM4379 and 17102-01 test reports.
- There are two contributing sulphide minerals in the five domains: chalcopyrite and pyrite. There is negligible to little other sulphide minerals in the ore.

Figure 13-3: Overall copper sulphide liberation

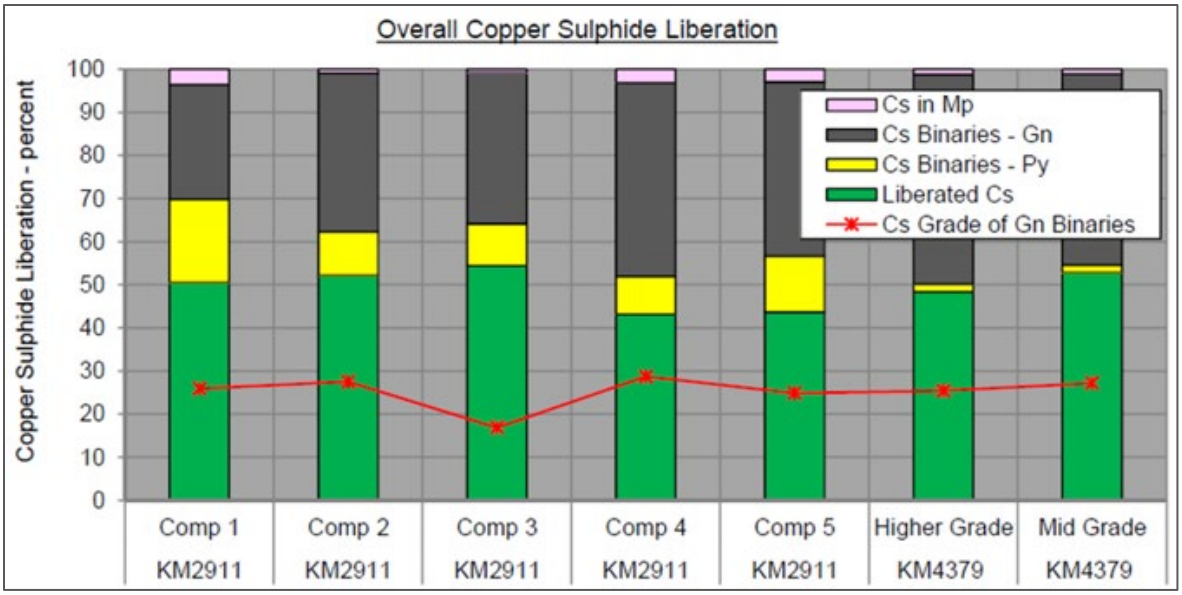
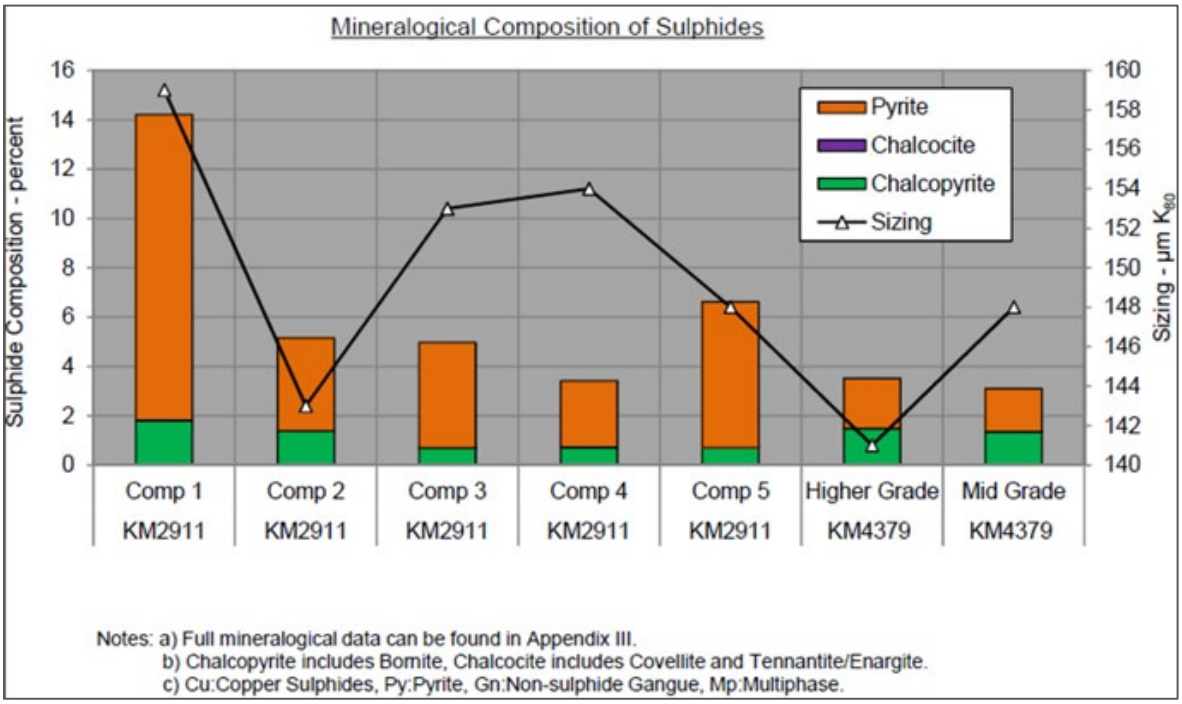


Figure 13-4: Sulphide mineralogy (ALS-KM4379)



The copper liberation data obtained through QEMSCAN Particle Mineral Analysis, at a P80 of 150 µm, is presented below:

- Chalcopyrite is either free or associated with non-sulphide gangue: 80–95%. The amount of chalcopyrite associated with pyrite is 5–20% shown in KM4379 while 17102-01 summarizes the value between 1.2% and 7.7%.

- Sulphide in the ore varies from 3% to 14% depending on the domain. The only domain that exceeds 6.5% sulphide is CP-1 at 14%.
- 80% of copper is fully liberated in the 20–30 µm size range. 50% of copper is liberated in the 30–50 µm size range.
- 80–90% of all pyrite is free or liberated across all domains. The remainder is predominantly complexed with other non-sulphide gangue.

Gold Mineralogy

The gold mineralogy size distribution at P80 of 150 µm is presented in Figure 13-5. The distribution of gold association is presented in Figure 13-6. The observations on gold mineralogy and liberation are summarized below:

- Gold is finely disseminated for all domains with 95-100% of the counts being finer than 10 µm; varying by CP domain.
- Gold association varies by domain:
- Gold was not associated with sulphides in large quantities (<20%) in domains CP-1 and CP-5. The other domains contained more gold associated and locked within silicates and sulphides (up to 80%).
- There was not much gold surface exposure with 20% exposed in CP-1; the remainder of the domains showed <10%.
- CP-1 and CP-5 showed native gold in some quantity (30% in CP-1 and 15% in CP-5). The remainder of the domains showed very little native gold.

It was noted in the 17102-01 summary recommendations that the gold mineral statistics were not ideal and that a full deportment study should be conducted if there were complications during the testwork program. KM4379 did not report the same observation.

The fine dissemination of gold and variable association with sulphides and silicates indicates that gold recovery may be more sensitive than copper recovery to changes in flotation operating strategy and cleaning selectivity. At the PEA level, this supports conservative gold recovery assumptions and further recommended testwork to refine gold deportment and recovery projections.

The high amount of gold locked with the sulphide is contributing to the high gold recovery from the proposed leach plant.

Figure 13-5: Gold mineralogy size distribution chart (SGS Report: 17102-01)

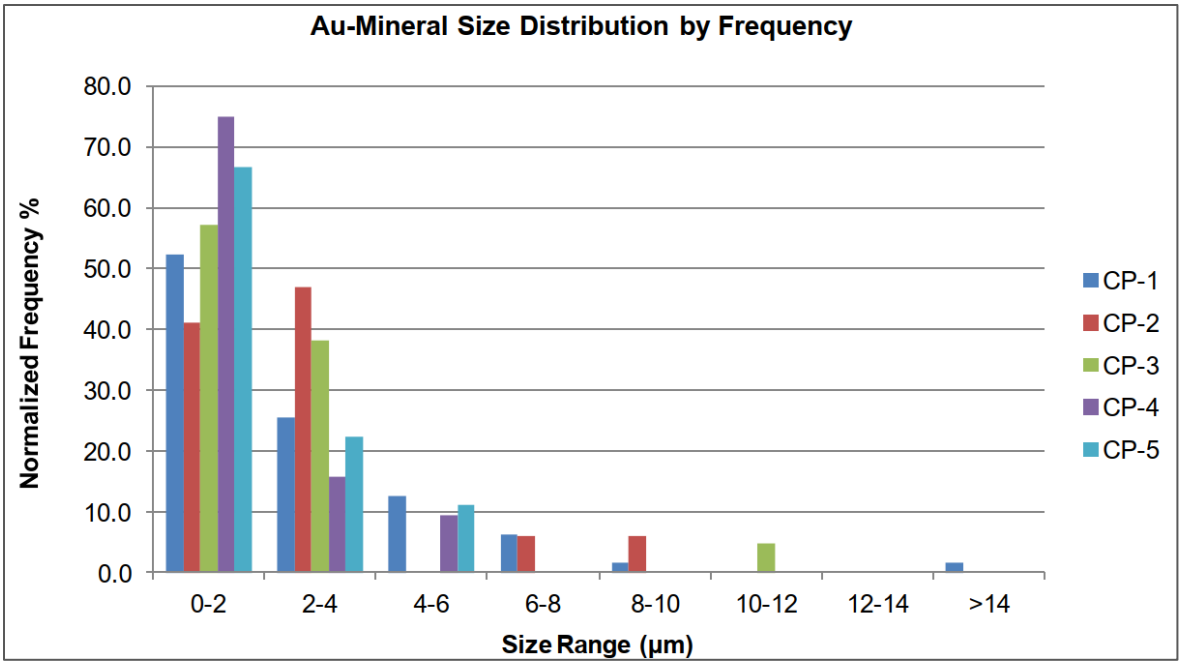
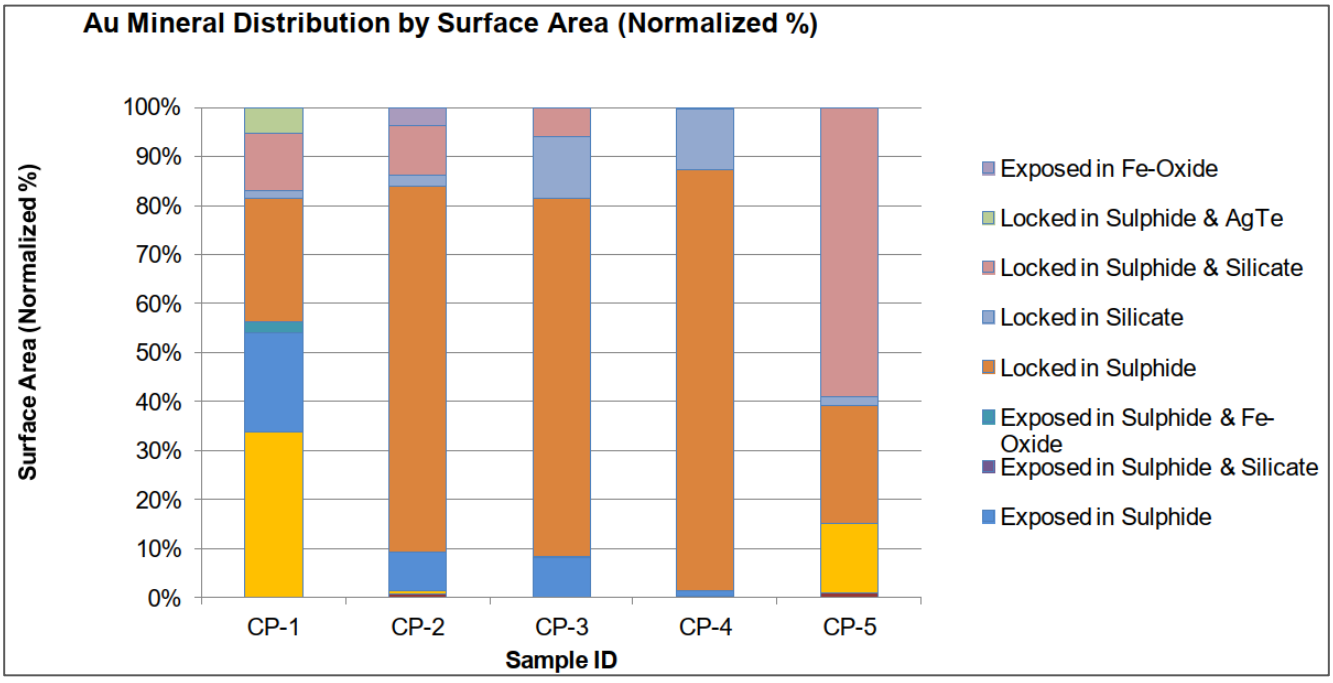


Figure 13-6: Gold mineral liberation (%) (SGS Report: 17102-01)



Comminution Characteristics

Specific Gravity

For the KUG CP-domain dataset, SG values range from approximately 2.26 to 2.91, with a mean of approximately 2.74 (n = 68). The 10th and 90th percentile SG values are approximately 2.60 and 2.86, respectively (Table 13-4).

Table 13-4: Summary statistics of SG tests on mineralized material

Parameters	Combined	CP-1	CP-2	CP-3	CP-4	CP-5
No. of samples	72	9	10	10	17	22
Average	2.74	2.82	2.70	2.77	2.68	2.76
Standard deviation	0.11	0.03	0.17	0.07	0.07	0.12
Coefficient of variation %	3.9%	1.1%	6.3%	2.4%	2.6%	4.2%
90% Confidence interval	0.8%	0.7%	3.7%	1.4%	1.1%	1.5%
P70/P50	1.04	1.01	1.01	1.01	1.00	1.01
P85/P50	1.04	1.02	1.01	1.01	1.02	1.04
Percentile values:						
Minimum	2.26	2.80	2.26	2.68	2.58	2.53
10%	2.60	2.80	2.55	2.70	2.60	2.60
20%	2.67	2.80	2.65	2.71	2.61	2.68
30%	2.70	2.80	2.69	2.72	2.66	2.70
40%	2.70	2.80	2.74	2.74	2.66	2.70
50%	2.76	2.80	2.78	2.78	2.70	2.79
60%	2.80	2.80	2.80	2.80	2.70	2.80
70%	2.80	2.82	2.80	2.80	2.70	2.80
80%	2.80	2.84	2.80	2.80	2.70	2.89
90%	2.85	2.86	2.80	2.81	2.80	2.90
Maximum	2.91	2.89	2.80	2.90	2.81	2.91

The SG range indicates moderate density variability across KUG domains. SG variability may influence classification performance, slurry density control, and comminution circuit stability depending on feed blend and operating conditions.

SMC Comminution Response

The SAG Mill Comminution (SMC) test is an abbreviated version of the standard JK drop-weight test performed on 100 rocks from a single size fraction (-22.4/+19.0 mm in this case). The SMC test results expressed as Drop Weight Index (DWI) are summarized in Table 13-5. The DWI for the Kemess Underground materials (72 samples) ranged from 4.13 kWh/m^3 (10th percentile) to 8.03 kWh/m^3 (90th percentile) and averaged 6.0 kWh/m^3. The KUG ores are characterized as moderate soft to hard with respect to resistance to impact breakage. The dataset shows a high degree of variability as measured by the coefficient of variance (27%), which compromises the treatment of the dataset as a single ore type.

Table 13-6 summarizes the statistics of the Axb values derived from the SMC tests. The Axb value varied from 33.9 (90th percentile) to 63.0 (10th percentile) and averaged 45.3.

Table 13-5: Summary statistics of the SMC DWI

Parameters	Combined (kWh/m³)	CP-1 (kWh/m³)	CP-2 (kWh/m³)	CP-3 (kWh/m³)	CP-4 (kWh/m³)	CP-5 (kWh/m³)
No. of samples	72	9	10	10	17	22
Average	6.03	4.77	5.62	6.30	5.37	6.90
Standard deviation	1.63	0.81	1.23	1.17	1.91	1.43
Coefficient of variation, %	27.1%	17.1%	21.8%	18.6%	35.6%	20.7%
90% Confidence interval	5.3%	10.6%	12.7%	10.8%	15.1%	7.6%
P70/P50	1.17	1.04	1.08	1.12	1.14	1.11
P85/P50	1.36	1.10	1.14	1.17	1.49	1.28
Percentile values:						
Minimum	2.08	3.56	3.42	4.22	2.08	4.45
10%	4.13	3.81	4.55	4.52	3.20	5.32
20%	4.57	4.23	5.04	5.51	3.74	5.74
30%	5.14	4.54	5.19	5.93	4.30	6.04
40%	5.47	4.68	5.35	6.20	4.83	6.43
50%	5.80	4.79	5.59	6.61	5.33	6.60
60%	6.41	4.87	5.75	6.96	5.51	7.16
70%	7.06	5.02	5.78	7.11	5.95	7.41
80%	7.44	5.12	6.08	7.27	7.18	8.22
90%	8.03	5.41	7.14	7.41	7.78	8.91
Maximum	9.85	6.40	7.90	7.53	9.04	9.85

Table 13-6: Summary statistics of the SMC Axb

Parameters	Combined (kWh/t)	CP-1 (kWh/t)	CP-2 (kWh/t)	CP-3 (kWh/t)	CP-4 (kWh/t)	CP-5 (kWh/t)
No. of samples	72	9	10	10	17	22
Average	45.31	59.55	45.74	42.28	49.99	39.60
Standard deviation	16.03	10.27	12.29	9.78	24.12	8.02
Coefficient of variation, %	35.4%	17.2%	26.9%	23.1%	48.3%	20.2%
90% Confidence interval	6.4%	10.5%	14.1%	12.5%	17.9%	7.1%
P70/P50	1.14	1.05	1.03	1.08	1.14	1.10
P85/P50	1.31	1.09	1.24	1.12	1.47	1.22
Percentile values:						
Minimum	127.16	78.37	59.33	60.50	127.16	52.71
10%	62.97	74.50	54.64	49.06	82.88	49.16
20%	58.82	68.69	53.11	46.73	70.18	46.00
30%	52.71	62.92	52.03	45.13	59.94	44.17
40%	48.97	61.39	50.52	42.55	54.89	43.03
50%	46.66	59.81	49.02	40.80	50.16	42.10
60%	43.06	58.68	48.98	39.37	48.16	39.11
70%	39.30	58.61	48.08	38.44	45.23	37.38
80%	37.24	56.08	40.76	37.92	37.25	35.20
90%	33.87	50.51	33.31	37.45	34.94	30.17
Maximum	28.22	43.10	32.20	36.50	31.43	28.22

Bond Ball Mill Work Index

Bond BWi values for the KUG CP-domain dataset range from approximately 9.9 kWh/t to 17.5 kWh/t, with a mean of approximately 14.0 kWh/t. (Table 13-7).

Table 13-7: Summary statistics of the Bond Ball Mill Work Index

Parameters	Combined (kWh/t)	CP-1 (kWh/t)	CP-2 (kWh/t)	CP-3 (kWh/t)	CP-4 (kWh/t)	CP-5 (kWh/t)
No. of samples	72	9	10	10	17	22
Average	14.34	13.37	13.59	14.28	13.56	14.75
Standard deviation	2.15	0.93	0.73	0.71	2.21	1.84
Coefficient of variation, %	15.0%	6.9%	5.4%	5.0%	16.3%	12.5%
90% Confidence interval	2.9%	4.3%	3.1%	2.9%	6.9%	4.6%
P70/P50	1.06	1.01	1.03	1.02	1.11	1.05
P85/P50	1.13	1.08	1.05	1.04	1.19	1.07
Percentile values:						
Minimum	9.87	12.24	12.30	13.39	9.87	10.43
10%	12.16	12.46	12.57	13.51	11.20	12.81
20%	12.79	12.59	13.07	13.63	11.59	13.69
30%	13.44	12.82	13.34	13.87	12.05	14.24
40%	13.72	13.12	13.47	14.12	12.78	14.65
50%	14.19	13.20	13.64	14.27	13.58	15.19
60%	14.48	13.30	13.88	14.35	14.22	15.42
70%	15.07	13.56	14.09	14.45	14.42	15.56
80%	15.54	14.10	14.19	14.67	15.21	15.74
90%	16.86	14.72	14.37	15.30	16.75	16.86
Maximum	22.38	14.91	14.52	15.55	17.34	17.53

The BWi distribution indicates moderate ball milling resistance overall with meaningful variability across domains. The upper end of the BWi range may result in increased grinding energy demand and grinding media consumption during harder ore periods.

Bond Abrasion Index

Bond Ai results were reported for a limited subset of KUG samples (n = 5), with values ranging from approximately 0.07 to 0.36 (mean approximately 0.16). (Table 13-8)

Table 13-8: Summary of Abrasion Index tests results

Sample name	Domain	AI (g)
KM2911 Composite 1	CP-1	0.36
KM2911 Composite 2	CP-2	0.08
KM2911 Composite 3	CP-3	0.12
KM2911 Composite 5 10-5	CP-5	0.17
KM2911 Composite 5 10-6	CP-5	0.07
Kemess 2019 Composite	Combined	0.19
Kemess 2019 Waste Comp	Hazelton	0.18

Sample name	Domain	AI (g)
Average Bond Abrasion Indices	0.17	
85th percentile Value	0.21	

The limited number of Ai results suggests abrasion characterization is incomplete across the full KUG variability dataset. The reported values indicate potential variability from mildly abrasive to moderately abrasive. At the PEA level, abrasion-related uncertainty should be addressed through conservative consumables assumptions and confirmed through additional testing in subsequent phases.

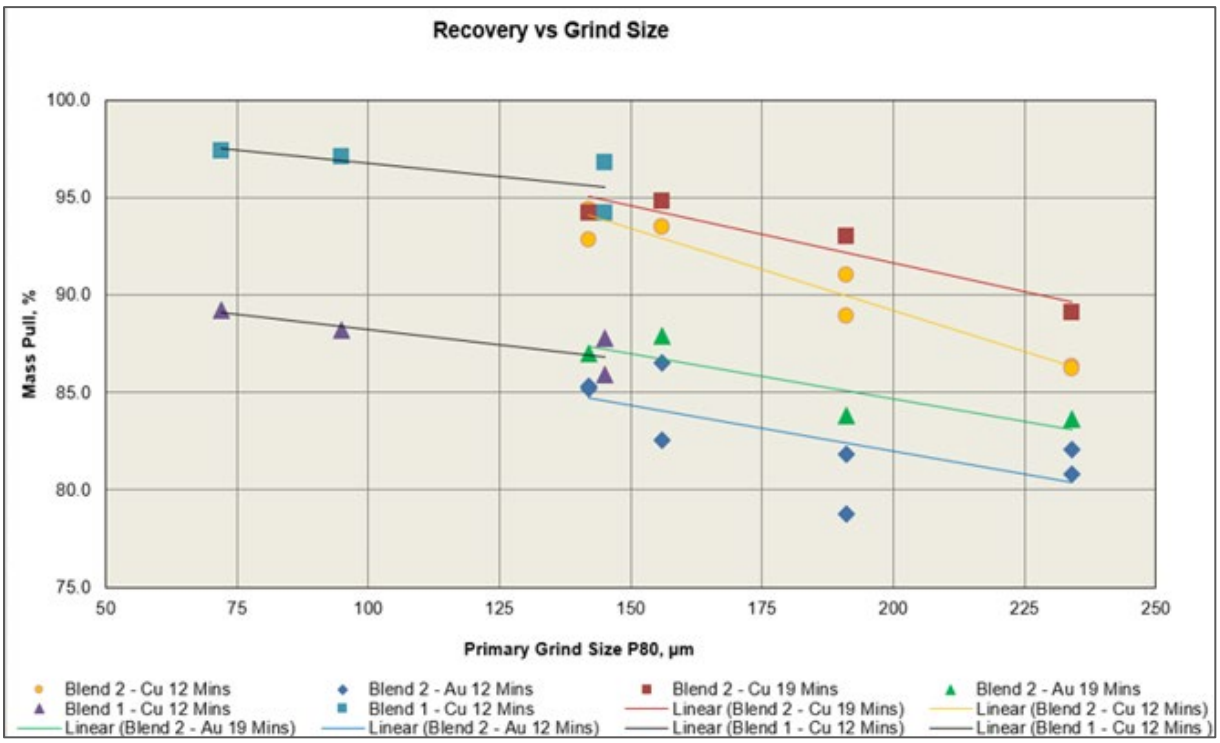
Flotation

Metallurgical flotation test work for the KUG material included assessments of primary grind size, rougher flotation performance, concentrate regrind requirements, reagent scheme development, and cleaner circuit response.

Primary Grind Size

Primary grind sensitivity tests indicated that copper and gold recoveries were generally not highly responsive within the range of grind sizes evaluated for both Blend 1 (high-grade) and Blend 2 (low-grade) composites. The primary grind size target of 150 µm as a design value was selected based on a high-level trade off study. The finer grinds tested showed only modest incremental recovery, which was insufficient to offset the additional energy and equipment requirements (Figure 13-7).

Figure 13-7: Mass recovery vs grind size



Rougher Mass Pull

Fifty-three flotation tests from the KM2911 program were reviewed across domains CP-1 through CP-5. Mass pull data summarized at the 85th percentile, presented in Table 13-9 indicated that CP-1 exhibited notably higher mass pulls, consistent with its higher sulphide content.

Table 13-9: Flotation tests results (KM2911)

Domain	85th percentile	# Tests (Open Rougher, Cleaner, and LCT)
CP-1	24.6	28
CP-2	13.4	5
CP-3	13.4	5
CP-4	12.7	5
CP-5	16.8	11
CP-6 (waste)	24.6	No Data
CP-7 (waste)	24.6	No Data

Optimization work completed in 2019 (Figure 13-8, Figure 13-9) showed that both copper and gold recovery continued to improve beyond 12 minutes of rougher retention time; however, the implications of extending residence time will require additional evaluation.

Figure 13-8: Copper recovery – 2019 KUG testwork

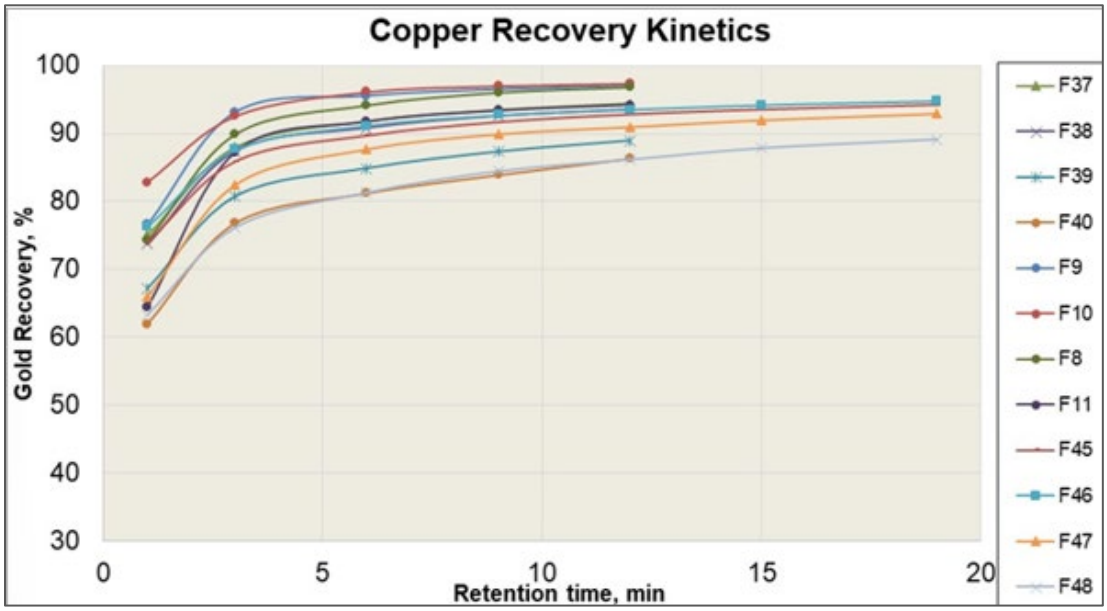
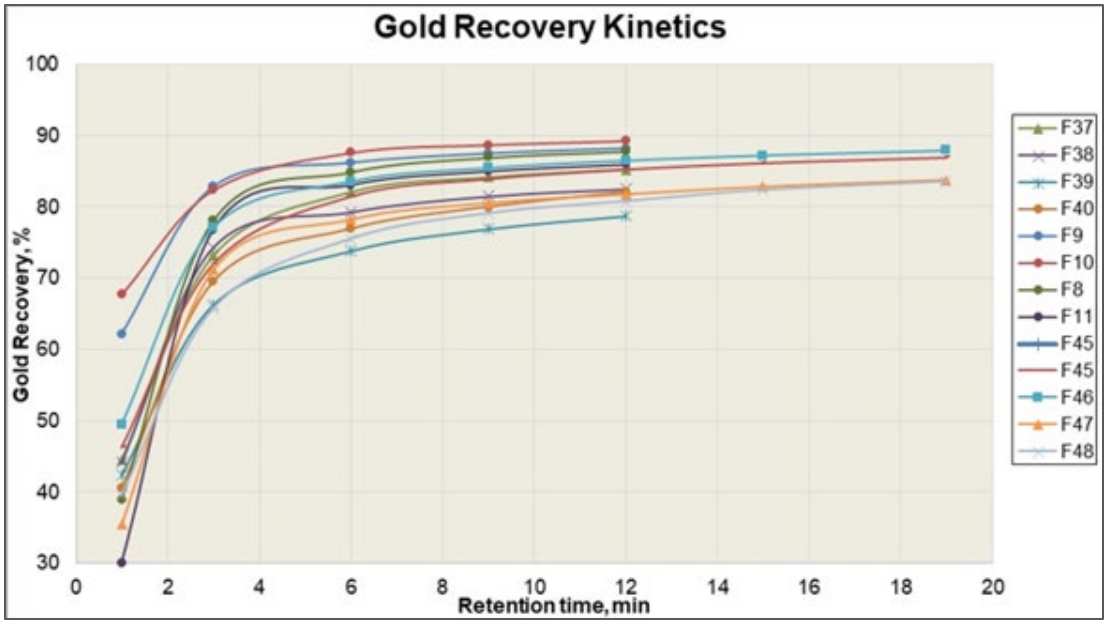


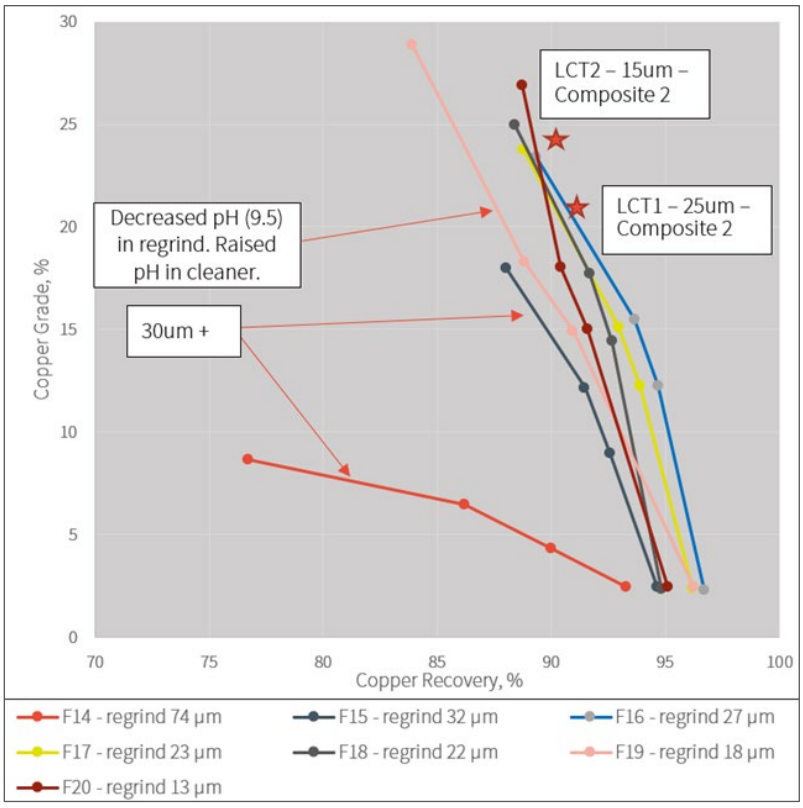
Figure 13-9: Gold recovery – 2019 KUG testwork



Concentrate Regrind Size

Regrind assessment from the 17102 testwork program showed that copper grade–recovery performance for blend composites remained relatively consistent between approximately 18–25 µm, with the strongest response observed near 27 µm. (Figure 13-10)

Figure 13-10: Blend composite grade recovery curves



Locked-cycle tests demonstrated only minor metallurgical differences when reducing the regrind size from 25 µm to 15 µm. At 25 µm, final copper concentrate grades remained above the design target of 21% Cu, with only small changes in overall recovery.

Locked-Cycle Testwork

Six locked-cycle tests were completed on composite blends made at various ratios between CP-1 to CP-7. These tests applied the primary grind (150 µm), regrind size (25 µm), and 19-minute rougher retention time established through previous optimization work. Final concentrate grades consistently exceeded the design target of 21% Cu. Locked-cycle results were used to develop metallurgical projection relationships, including a copper recovery curve capped at 95% and a linear gold recovery relationship with head grade.

13.2.2 Metallurgical Testwork Program (Main Zone)

Sample Selection

In 2025, ALS Metallurgy completed a metallurgical testwork program (KM7413) to evaluate the performance of Main Zone material. Approximately 4 t of ¼ and ½ HQ drill core from the 2024 drilling campaign were collected to provide representative feed for the program. From this material, 30 metallurgical samples were prepared and subsequently combined into composite samples designed to reflect key Main Zone ore types.

Composite construction focused on the two principal geological domains: Broken Zone and Not Broken Zone material. These composites were assembled from selected metallurgical samples to capture the variability within the Main Zone and to support the development of metallurgical response predictions for the current study. An additional composite representing early production years (Years 1–4) was also generated to provide indicative guidance on early plant performance.

Head Assay

Early ML composite and composite samples from the Main Zone and Underground deposits were submitted for screen metallics, chemical assay analysis, and an ICP multi-element scan to characterize the copper, gold, sulphide, arsenic and other elemental content.

Early ML composite, made up of 76% Broken and 24% Not-Broken, reported assay values with 0.18% Cu and 0.36 g/t Au. Assay values for the samples were found to be in the range of 0.16% to 0.21% Cu and 0.32 g/t to 0.42 g/t Au. The results are summarized in Table 13-10.

Table 13-10: Head grade characterization summary

Element/Sample ID	Main Zone		
Unit	**Broken**	**Not Broken**	**Early ML**
Au g/t (SM)	0.42	0.32	0.36
Cu %	0.21	0.16	0.18
As %	<0.001	<0.001	<0.001
C(t) %	0.01	0.18	0.12
C(g) %	---	---	---
TOC leco %	---	---	---
CO_3 %	---	---	---
S %	1.88	3.76	1.84
$S^=$ %	1.81	2.27	1.81
SO_4 %	0.08	1.49	0.03
S° %	---	---	---

The head grade ranges indicate variability in copper, gold, and sulphide content across the two domains. This variability is expected to influence flotation response, mass pull, and concentrate quality outcomes. At the PEA level, the results support conservative recovery assumptions and ore-type dependent metallurgical interpretation.

Mineralogy

Mineralogical characterization for the Main Zone material was completed using data from the most recent testwork programs, including particle mineral analysis by QEMSCAN Particle Mineral Analysis protocols and size-by-liberation datasets available for Broken, Not Broken, and early mine life material. These data were evaluated at a nominal primary grind size of approximately P80 of 150 µm.

Copper Mineralogy

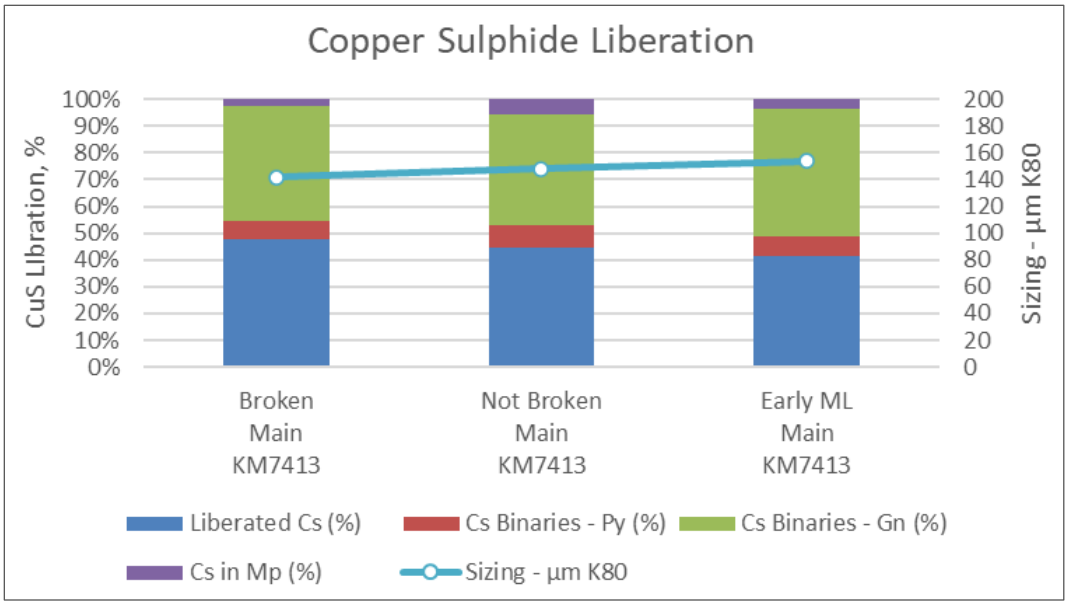
Across Broken, Non-Broken Zone and Early Mine Life samples, chalcopyrite is the primary copper-bearing mineral. Minor secondary copper sulphides and small fractions of non-sulphide copper were identified depending on the composite.

All three samples contain significant pyrite, and pyrite-to-copper sulphide ratios ranging from ~5:1 to 11:1 were reported across the main composites. This mineralogical context highlights the need for strong selectivity and pyrite rejection in cleaner flotation.

Copper Liberation

Copper sulphide liberation for the main composite suite – representing Broken, Not Broken, and Early Mine Life, ranging between 41% and 48% at the nominal primary grind size. Non-liberated copper sulphides were primarily associated with non-sulphide gangue, rather than pyrite. This is presented in Figure 13-11. The size fraction distribution of the copper is presented in Table 13-11.

Figure 13-11: Overall copper sulphide liberation (KM7413)



Pyrite is generally well liberated at the test grind around P80 of 150 µm, with remaining non-liberated pyrite occurring mainly in binary association with non-sulphide gangue.

Table 13-11: Copper sulphide mineral particle passing (KM7413)

Cu – Sulphides Diameter at % Passing in µm	Broken	Not Broken	Early ML
80% Passing	32	33	33
50% Passing	32	33	33
20% Passing	82	81	83

The dominance of chalcopyrite and the high degree of chalcopyrite liberation at the nominal grind size support the selection of a conventional sulphide flotation flowsheet. Chalcopyrite association with pyrite indicates that cleaner flotation selectivity may influence both copper and gold recovery, particularly where pyrite rejection is required to achieve concentrate quality targets

Comminution Characteristics

Comminution testwork for the three Main Zone samples included Bond low-impact (crusher) tests, Bond Ball Mill Work Index (BWi), Bond Abrasion Index (Ai), and SMC testing; however, the amount of work completed to date is limited compared to the Kemess Underground program and represents only a preliminary assessment of Main Zone hardness characteristics. The SMC results yielded A × b values ranging from 40 to 69, indicating material that ranges from moderately soft to hard with respect to impact breakage, but the current dataset is insufficient to evaluate variability across the Main Zone. Additional comminution testing is recommended in the next phase to better define grindability, capture domain variability, and support development of robust design criteria for the Main Zone (Table 13-12).

Table 13-12: Comminution parameters (testwork average values)

Metric (average)	Units	Main Zone		
		Broken	Not Broken	Early ML
No. of samples	-	1	1	1
Ore SG	-	2.66	2.75	2.61
SMC DWi	kWh/m^3	3.81	6.56	4.78
SMC Axb	kWh/t	69.1	40.7	54.5
BWi	kWh/t	17.8	16.1	17.3
Ai	-	0.02	0.05	0.03

Comminution testwork indicates clear differences in breakage response between the two Main Zone ore types. SMC results show that Not Broken Zone material is harder, with lower A×b values and correspondingly higher simulated specific energy requirements, implying greater SAG mill energy demand or reduced throughput when this material is dominant. Bond Ball Mill Work Index results reflect moderate to hard resistance to grinding for both ore types, indicating that ball mill energy requirements may be significant, while abrasion index values indicate mildly abrasive behaviour. Overall, the dataset confirms that comminution energy demand and circuit performance will vary with the relative proportions of Broken and Not Broken material, and additional variability-based comminution testing is recommended in the next phase to refine the milling circuit design.

Comminution testwork indicates differences in breakage response between the two Main Zone samples. SMC results show that Not Broken Zone material is harder, with lower A×b values and correspondingly higher simulated specific energy requirements, implying greater SAG mill energy demand or reduced throughput when this material is dominant. Bond Ball Mill Work Index results reflect moderate to hard resistance to grinding for both ore types, suggesting non-trivial ball mill energy requirements, while abrasion index values indicate mildly abrasive behaviour.

Overall, the dataset confirms that comminution energy demand and circuit performance will vary with the relative proportions of Broken and Not Broken material in the Main Zone, and additional variability-based comminution testing is recommended in the next phase to refine the milling circuit design.

Flotation

Sulphide flotation testwork was conducted on the Broken, Not Broken, and Early Mine Life composites to evaluate rougher and cleaner performance, assess reagent requirements, examine grind-size sensitivity, and confirm flowsheet performance through locked-cycle testing. In addition to the composite-level program, rougher and cleaner tests completed on individual variability samples provided further evidence of performance differences between ore types and helped characterize the range of metallurgical response expected from the Main Zone material.

Rougher Flotation

Kinetic rougher tests were completed at nominal primary grind targets of approximately 100 µm, 150 µm, 200 µm, and >300 µm P80. Copper recovery to rougher concentrate showed limited sensitivity between 100–200 µm, averaging ~91%, ~90%, and ~88% respectively. Gold recovery remained similarly stable at ~82–83% across this range, indicating that gold recovery is driven largely by pyrite deportment rather than chalcopyrite liberation. At coarser primary grinds (>300 µm), rougher performance declined, with recoveries decreasing to approximately 77% for copper and 74% for gold.

Cleaner Flotation

Cleaner flotation optimization evaluated the influence of cleaner pH, collector dosage, and regrind sizing on concentrate grade and recovery. Testing showed that operation near pH 11 produced the most consistent selectivity, while lower pH resulted in reduced copper and gold recovery. Higher collector dosages generally improved metal recovery but often reduced concentrate grade through increased pyrite entrainment, underscoring the need to balance reagent addition with concentrate quality objectives. Regrind sizing had composite-dependent effects: finer regrind improved recovery but increased the risk of pyrite carryover, whereas coarser regrind enhanced grade in certain cases. Limited trials with CMC indicated that non-sulphide gangue depression may be beneficial under specific conditions but is not universally required. Overall, the cleaner testwork supports a flowsheet incorporating two-stage cleaning with controlled reagent addition and regrind sizing to maintain selectivity while maximizing overall recovery.

Locked-Cycle Testwork

A locked-cycle test was completed for each Main Zone composite using conditions defined during cleaner optimization. All locked cycle tests employed a two-stage cleaning configuration, which consistently delivered the best balance of copper grade and recovery. Across composites, copper concentrate grades averaged approximately 23% Cu, with copper recovery of ~88.5% (Figure 13-13) and gold recovery of ~58.5% (Figure 13-12). Gold losses to first-cleaner tails ranged from 15–30%, reflecting the dominant association of gold with pyrite. This is an observation that is consistent with mineralogical data for these composites.

Figure 13-12: Gold recovery – 2025 Main Zone testwork

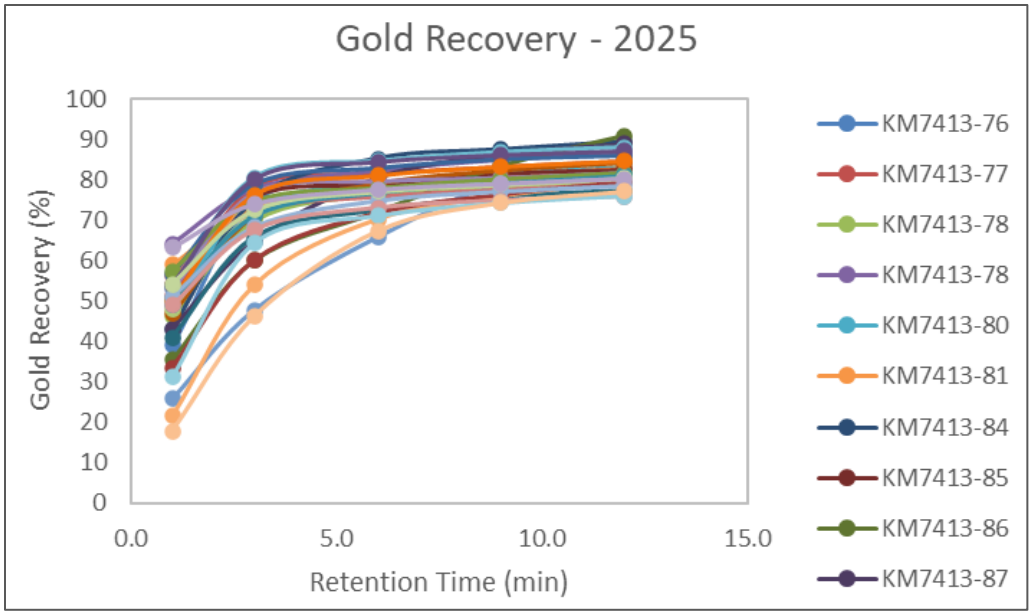
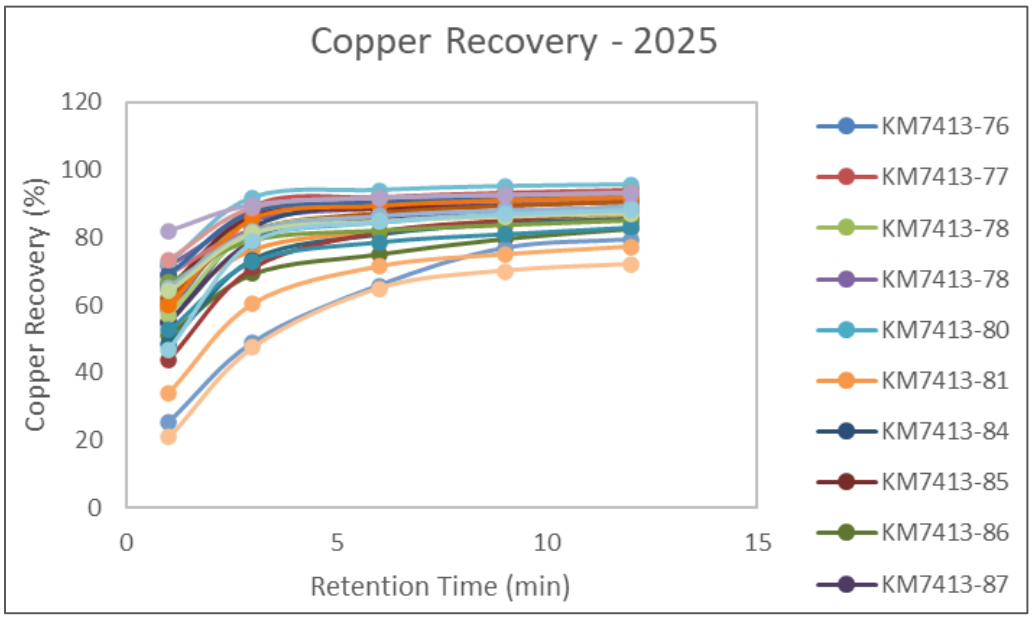


Figure 13-13: Copper recovery – 2025 Main Zone testwork



The graphs above illustrate the consistent upward trend in recovery beyond 12 minutes across both datasets, confirming that extended retention time is critical for achieving optimal performance.

13.3 HISTORICAL OPERATING DATA

The historical Kemess concentrator operated on two parallel SAB circuit consisting of two 34' × 13.5' SAG mill (6000HP motor) and two 22' × 36.5' ball mill (6000HP motor). Across its operating life (1998 - 2011), the plant consistently processed ~50 to 52 ktpd, corresponding to approximately 18.6 Mtpa of ore, with the primary grinding typically at P80 of 150 µm.

Figure 13-14, Figure 13-15 and Figure 13-16 illustrate historical performance trends for plant throughput (including a reference line indicating the design annual throughput Mtpa), metal head grades, and recovery.

Figure 13-14: Plant throughput – Historical plant data

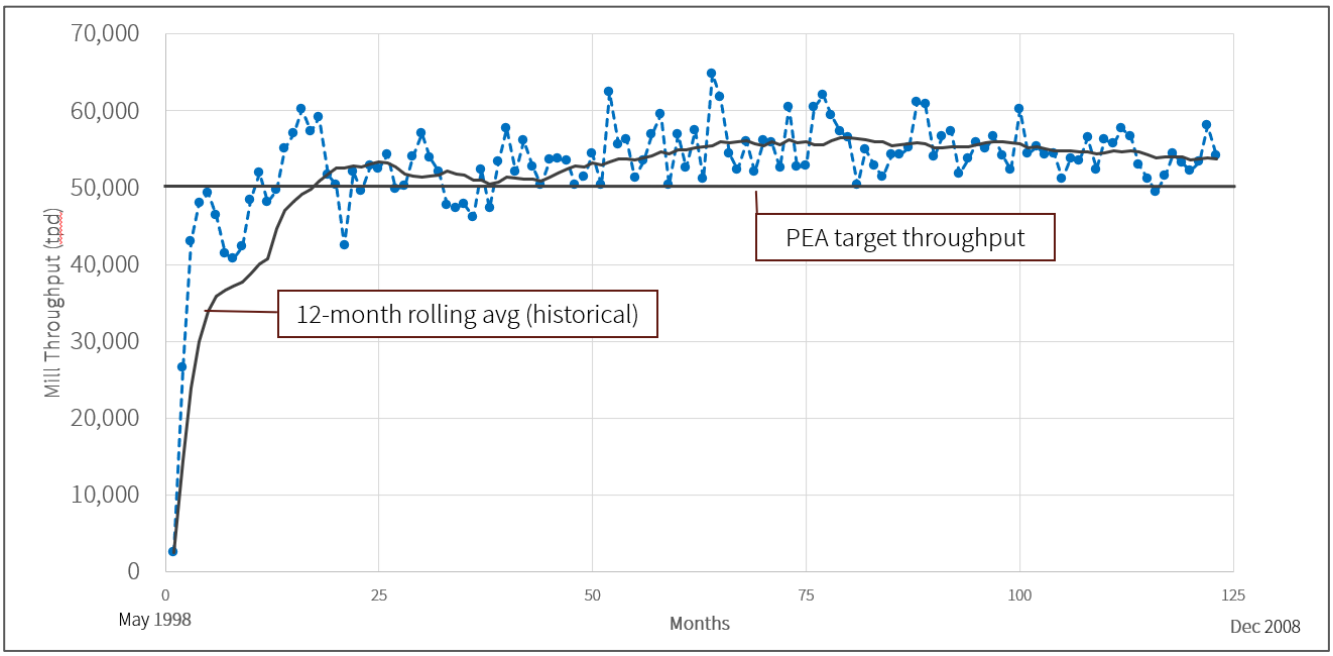


Figure 13-15: Metals head grades – Historical plant data

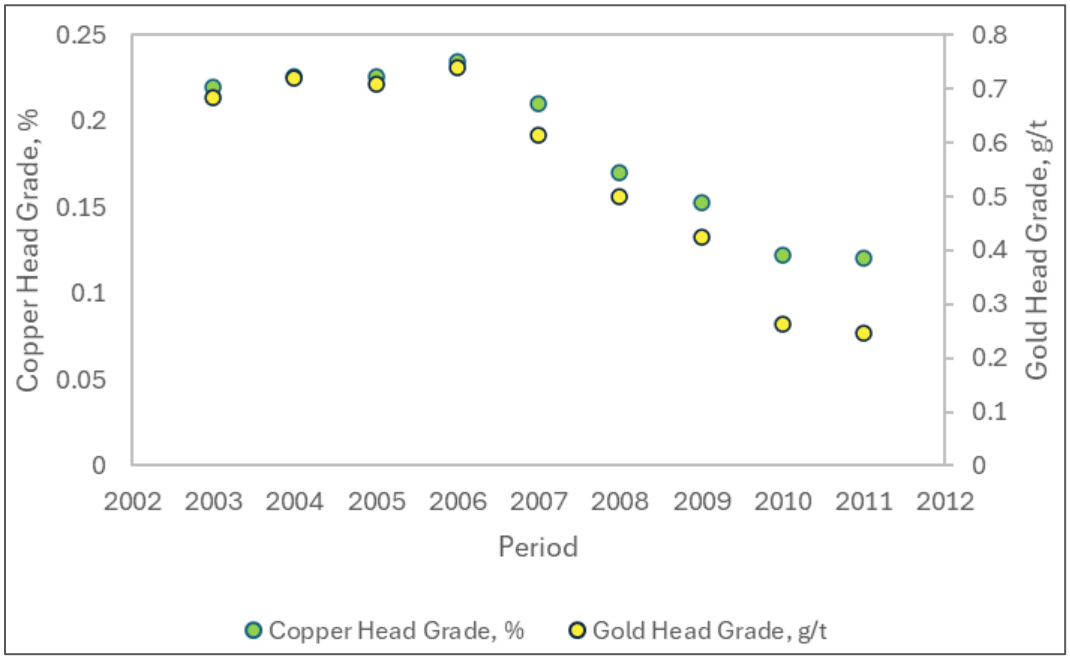
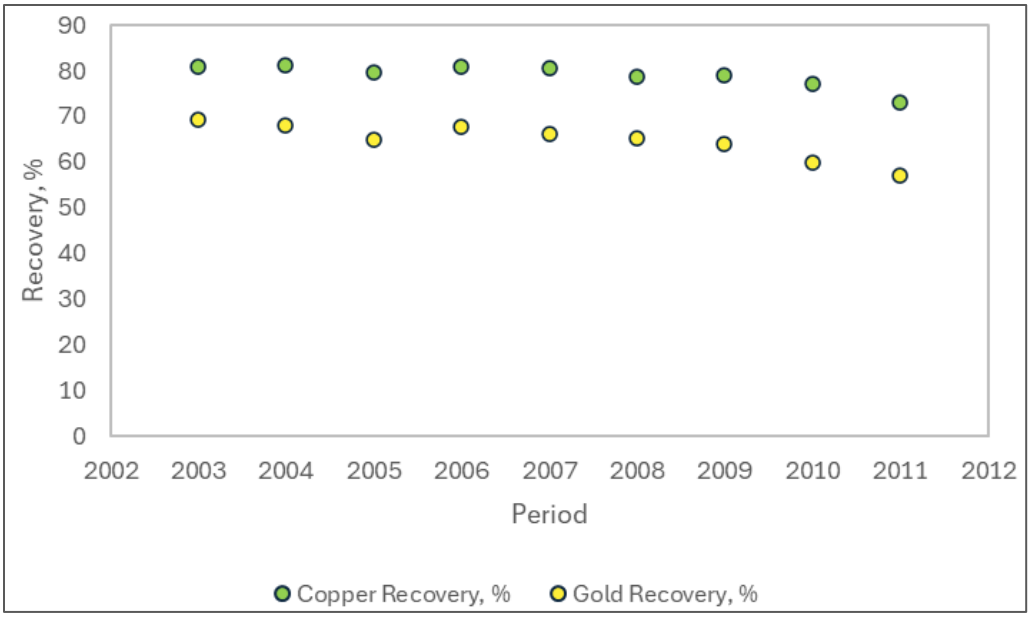


Figure 13-16: Metals recovery – Historical plant data



While the historical ore type processed at Kemess South differs from the Main Zone and KUG material evaluated in this PEA, the long operating record remains highly valuable. It provides confidence that the existing mill infrastructure is capable of reliable, high-availability operations, supporting the viability of the refurbishment and expansion strategy of this PEA. Table 13-13 shows the comparison of key comminution characteristics between historical ore body and Main Zone/KUG. This drives the flowsheet modifications, as detailed in Item 17.

Table 13-13: Comminution parameters (Main Zone vs Underground, average values)

Metrics	Units	Main Zone			Underground					Kemess South (historical)
		Broken	Not Broken	Early ML	CP-1	CP-2	CP-3	CP-4	CP-5	
No. samples	-	1	1	1	9	10	10	17	22	-
Ore SG	-	2.66	2.75	2.61	2.82	2.70	2.77	2.68	2.76	-
SMC DWi	kWh/m^3	3.81	6.56	4.78	4.77	5.62	6.30	5.37	6.90	-
SMC Axb	kWh/t	69.1	40.7	54.5	59.55	45.74	42.28	49.99	39.6	44.6–79.0
BWi	kWh/t	17.8	16.1	17.3	13.37	13.59	14.28	13.56	14.75	13–16.5
Ai	-	0.02	0.05	0.03	0.36	0.08	0.12	0.17	0.07	0.03

13.4 DELETERIOUS ELEMENTS

13.4.1 Deleterious Elements in Feed

A total of 30 head samples were analysed using a multi-element ICP scan from Main Zone. The results indicate that key penalty elements commonly monitored for copper concentrates (e.g. As, Sb, Bi, Hg, Pb, Zn, Se, Te) are present in the feed at low ppm levels, with variability across the dataset.

Table 13-14 summarizes the key deleterious element statistics for the head (feed) samples.

Table 13-14: Summary of selected deleterious elements – Feed assays

Element	Unit	Minimum	P10	P50	Mean	P90	Maximum
As	ppm	0.9	1.9	3.75	3.69	5.5	7.1
Sb	ppm	0.32	0.55	1.07	1.15	1.69	3.75
Bi	ppm	0.16	0.27	0.66	0.71	1.09	2.7
Hg	ppm	0.005	0.005	0.007	0.012	0.015	0.071
Pb	ppm	4.8	6.8	11.6	13.7	16.3	65.1
Zn	ppm	76.0	105.5	135.5	149.9	199.5	348
Se	ppm	1.00	3.00	3.50	3.56	5.00	6.00
Te	ppm	0.15	0.24	0.43	0.48	0.78	1.11
Cd	ppm	0.13	0.38	0.63	0.77	1.29	2.27

Overall, arsenic, antimony, bismuth, and mercury were detected at low ppm levels in the feed samples. Selenium and tellurium were detected at low ppm levels with limited variability. Lead and zinc show higher absolute ppm values than other potential penalty elements.

Deleterious Elements in Concentrate

Review of the locked cycle testwork for the Broken Zone and Not Broken Zone concentrates indicates that most deleterious elements occur at levels well below typical smelter penalty thresholds. However, a small number of constituents may require operational awareness or commercial consideration. Table 13-15 summarizes the key deleterious element statistics for the concentrate samples.

Table 13-15: Summary of selected deleterious elements – Concentrate assays

Element	Unit	Broken Zone	Not Broken Zone	Early ML
Sb	g/t	7	37.7	6.9
As	g/t	15	20	19
Bi	g/t	26.5	28	24.3
Cd	g/t	57.8	51	68.4
Cl	g/t	100	150	130
Cu	%	22.9	25.3	20.1
F	g/t	340	170	370
Au	g/t	29.9	34.1	27.8
Fe	%	25.9	30.1	26.2
Pb	%	0.04	0.05	0.09
Mg	%	0.86	0.35	0.86
Hg	g/t	0.32	0.19	0.57
Mo	%	0.4	0.86	0.49
Re	g/t	10.2	25.7	14.55
SiO_2	%	10.1	4.5	11.6
Ag	g/t	140	131	96
S	%	31.6	35.2	31
U	g/t	1	1	1
Zn	%	0.15	0.06	0.26

Within the Broken Zone and Not Broken Zone concentrates, silica from the Broken Zone concentrate is elevated; blending may be required with lower-silica domains to maintain marketability. Overall, the deleterious-element profile for these domains is manageable, with no major constraints anticipated.

13.5 LEACH TESTWORK

The proposed feed to the leach circuit is the cleaner-scavenger tails. In August 2025, ALS Metallurgy performed preliminary cyanide leach testwork on six different samples to determine the leachable gold recovery of the first cleaner tails stream. The samples used for this leaching work were from locked-cycle flotation tests. This testing provides a baseline of expected gold recovery without any pre-treatment steps.

Table 13-16 summarizes the first cleaner tails composition and elemental recoveries from the locked-cycle tests.

Table 13-16: Locked cycle flotation composition and recoveries of first cleaner tails

No.	Locked cycle test sample	Weight % of feed	Assay (% or g/t)				Recovery (%)			
			Cu	Fe	S(t)	Au	Cu	Fe	S(t)	Au
1	KM7413-82 Nugget	10.0	0.05	19.4	19.68	1.02	3.8	27.8	85.8	22.7
2	KM7413-83 Broken Zone	9.3	0.200	15.6	16.45	1.45	8.5	26.3	83.3	30.3
3	KM7413-92 Not Broken Zone	9.9	0.05	18.1	18.8	0.70	2.8	26.4	49.6	20.9
4	KM7413-93 Year 1-4	10.4	0.12	15.6	14.8	1.0	7.0	25.4	83.2	28.4
5	KM7413-94 Year 5-8	9.7	0.04	18.8	16.8	0.79	2.4	26.2	50.8	21.9
6	KM7413-95 Year 9-EOL	10.2	0.03	15.7	16.6	0.47	1.7	27.1	42.4	15.3

Based on the locked cycle flotation test results, the leach feed should contain between 15% and 20% sulphur, at gold grades between 0.47 g/t and 1.45 g/t. The SGS mineralogical composition analysis of the mill feed showed that more than 98% of the sulphur is present as sulphide.

13.5.1 Cyanide Leach Tests

ALS performed cyanide leach bottle roll tests on six different samples of copper first cleaner tails. The leach testwork feed samples were produced from locked-cycle flotation testwork, which included a rougher concentrate regrind stage, producing a leach feed grind P80 size between 19 µm and 22 µm. Leach test conditions were with 500 ppm NaCN at pH 11.0, with oxygen sparging to the head space. All tests were performed on 500 g sample at 40% solids.

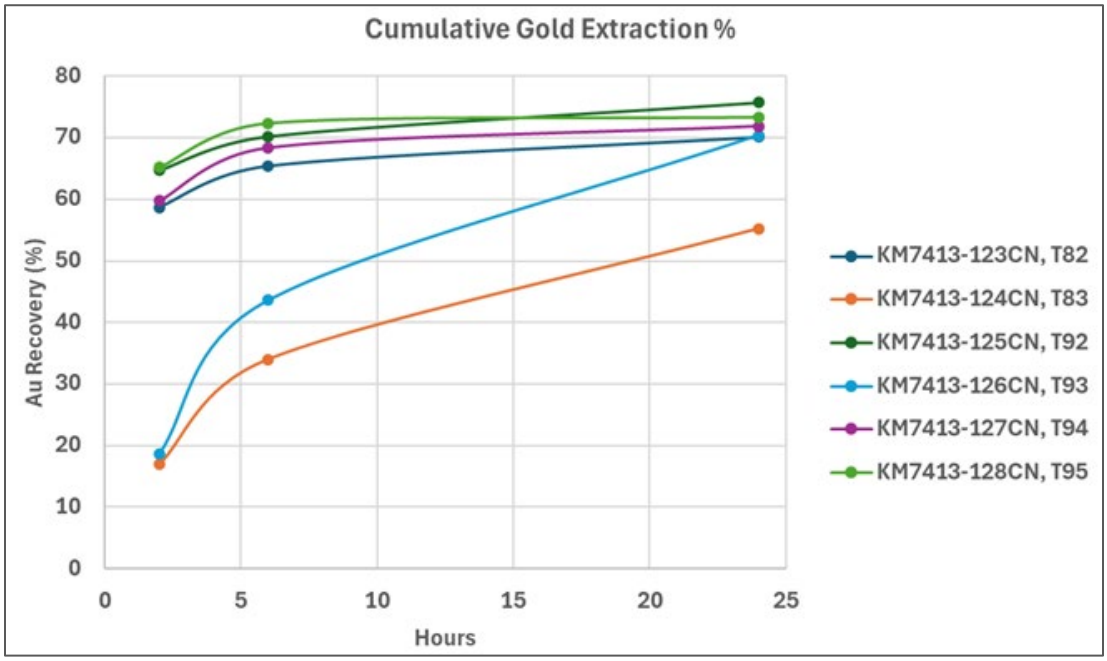
Sample feed measured gold grades ranged from 0.49 g/t to 1.31 g/t. The leach test results indicate gold recoveries of 55% to 76% achieved. The results of the leach tests are summarised in Table 13-17.

Table 13-17: Cyanide leach tests results summary

Test	Sample	Nominal P80 (µm)	Feed grade		Reagent consumption		Au recovery at 24 h (%)	Solid tail assay		
			Meas (g/t)	Calc (g/t)	NaCN (kg/t)	Lime (kg/t)		Cut 1 (g/t)	Cut 2 (g/t)	Avg (g/t)
KM7413-123CN	T82 Cu1CT	19	0.93	0.77	0.5	2.7	70.1	0.24	0.22	0.23
KM7413-124CN	T83 Cu1CT	22	1.31	1.15	1.9	2.1	55.2	0.51	0.52	0.515
KM7413-125CN	T92 Cu1CT	22	0.67	0.66	0.7	3.2	75.7	0.16	0.16	0.16
KM7413-126CN	T93 Cu1CT	21	1.05	0.96	1.5	1.9	70.4	0.27	0.3	0.285
KM7413-127CN	T94 Cu1CT	22	0.67	0.59	0.8	2.3	71.9	0.17	0.16	0.165
KM7413-128CN	T95 Cu1CT	22	0.49	0.49	0.6	1.8	73.3	0.13	0.13	0.13

Leach kinetic results indicate fast leaching rates during the first 6 hours on four of the six tests, shown in Figure 13-17. Two of the samples, T83 (“Broken Zone”) and T93 (“Year 1-4”), showed slower leach rates with higher cyanide and lower lime consumption compared to the other four samples. These slower leaching samples contain higher copper content at 0.2%wt. for T83 and 0.12%wt. for T93, indicating the presence and impact of cyanide leachable copper.

Figure 13-17: Gold leach kinetics of samples from copper first cleaner tails



The recovery of the leach circuit has been assumed to be 70% for the PEA and further test work is planned to future phases of the project.

13.6 RECOVERY ESTIMATES

Recovery estimates for this project are developed from locked cycle testwork completed on Main Zone composites in the 2025 program and locked cycle test results for KUG completed in the 2019 program. Main Zone recoveries were derived from master composites representing Broken Zone and Not-Broken

Zone material. For KUG, recovery estimates were developed using available 2019 LCT results for blended composites and individual geometallurgical domains.

13.6.1 Main Zone Recovery Estimates

Locked cycle results for Main Zone Broken and Not Broken composites are summarized in Table 13-18.

Table 13-18: Main Zone locked cycle test results summary (2025 program)

Composite	Primary grind P80 (µm)	Regrind grind P80 (µm)	Cu head (%)	Au head (g/t)	Cu rec (%)	Au rec (%)	Cu conc (%)	Au conc (g/t)
Broken Zone	142	22	0.22	0.44	82.9	53.5	22.9	29.9
Not Broken Zone	148	22	0.16	0.33	89.9	58.8	25.3	34.1

Copper recoveries for Main Zone composites range from approximately 82.9% to 89.9%, while gold recoveries range from approximately 53.5% to 58.8%. The testwork produced a copper concentrate grading approximately 23–25% Cu, which is considered reasonable for PEA-level assumptions.

Main Zone recoveries were applied only to material mined by open pit.

13.6.2 Kemess Underground Recovery Estimates

In 2019, AuRico determined five main geometallurgical domains from the KUG feasibility study. For the metallurgical testing programs, blended and individual composite samples were tested in two separate campaigns with the locked cycle test results summarized in Table 13-19.

Table 13-19: Locked cycle test summary (SGS 2019)

SGS 2019	Composite	Cu grade (%)	Au grade (ppm)	Cu recovery (%)	Au recovery (%)
Campaign 1	Blend 1	0.32	0.78	94.0	70.3
	Blend 2	0.28	0.63	93.1	69.2
	Blend 3	0.31	0.71	94.6	70.4
	Blend 4	0.3	0.71	94.2	69.1
Campaign 2	CP-1	0.57	1.57	95.7	73.4
	CP-2	0.20	0.42	89.8	72.7
	CP-3	0.28	0.64	91.9	71.0
	CP-4	0.23	0.38	88.6	68.0
	CP-5	0.23	0.45	90.5	51.5

Based on the data, the following regression equations were proposed for copper and gold recoveries. These equations can be applied to the underground mineralized material.

- Copper recovery = $101.26\ x^{0.0733}$, where x is copper head grade
- Gold recovery = $7.3536\ x + 63.261$, where x is gold head grade.

13.6.3 Kemess Leach Plant Recovery Estimates

The results of the leach testwork shows that roughly 70% of the gold in the cleaner scavenger tailings is recovered into doré. For this study, it was assumed that the average amount of gold in the cleaner scavenger tailings was 20% and the leach plant would recover 70% of this, resulting in a global recovery increase of 14%. Therefore, when the leach plant comes online, the recoveries of both the Main Zone and Underground will increase by 14%.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The Kemess Mineral Resource estimate presented herein (the “MRE”) was prepared by Centerra technical staff in 2025. The effective date of the Kemess Project MRE is December 31, 2025.

The MRE is a reasonable representation of the global gold mineral resources of the Project at the current level of sampling. It has been completed in conformity with the widely accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines (CIM, Nov 2019) and is reported in accordance with the Canadian Securities Administrators’ National Instrument 43-101 Form F1. The resource includes four deposits – two open pits – Kemess Main and Kemess South, and two underground deposits – Kemess Underground (KUG), and Kemess East.

The modelled mineralization of the Kemess Main zone, which comprises Kemess Main Open Pit, KUG and Kemess East, measures 3.9 km along strike, 0.6 km wide, and 1.7 km deep. Excluding Kemess East, the Kemess Main Zone is 0.8 km deep. The MRE presented herein considers all current lithological and alteration data to support the statistical evaluation of gold, copper, and silver assays. The model incorporates exploration drill hole data up to March 24, 2025, and utilizes gold mineralization domains that are based on a 0.2 g/t Au grade, copper mineralization domains based on a 0.2% Cu grade, and silver mineralization domains based on 1.5 g/t Ag grade. Mineralization is largely hosted in the Black Lake lithological unit.

At Kemess South, the resource area measures 2.1 km along strike, 0.7 km across strike, and is 0.5 km deep. The model incorporates exploration drillhole data up to November 18, 2025, and utilizes gold mineralization domains that are based on a 0.2 g/t Au grade, copper mineralization domains based on a 0.2% Cu grade, and silver mineralization domains based on 1 g/t Ag grade. Like the Kemess Main Zone, mineralization is largely hosted in the Black Lake lithological unit.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves.

The following section documents the modelling and statistical approach that was used for the completion of the Kemess Main Zone and Kemess South resource models, and estimation of mineral resources.

14.2 METHODOLOGY

Leapfrog Geo (v.2024.1.2) along with the Leapfrog Edge extension was used to construct the domain solids, to prepare assay data for geostatistical analysis, construct the block model, estimate gold grades, and tabulate mineral resources.

The evaluation of mineral resources involved the following procedures:

- Database compilation and verification
- Construction of geological and major mineralized domains
- Definition of geostatistical resource domains
- Data conditioning (compositing and capping) for geostatistical analysis
- Variography
- Selection of estimation strategy and estimation parameters
- Block modelling and grade interpolation
- Validation, classification, and tabulation
- Assessment of “reasonable prospects for eventual economic extraction”
- Selection of reporting assumptions
- Preparation of the Mineral Resource Statement

The following sections summarize the methodology and assumptions made by Centerra to construct the mineral resource model.

14.3 DRILLHOLE DATABASE

The global Kemess database comprises 907 diamond drillholes (338,304 m), yielding 159,897 gold, 159,928 copper, and 136,311 silver assays.

The effective date of the Kemess Main Zone drilling database is March 24, 2025. The 2025 Kemess Main Zone MRE utilises 417 drillholes (247,763 m), excluding 468 drillholes (88,302 m) as they are outside of the main mineralized domains.

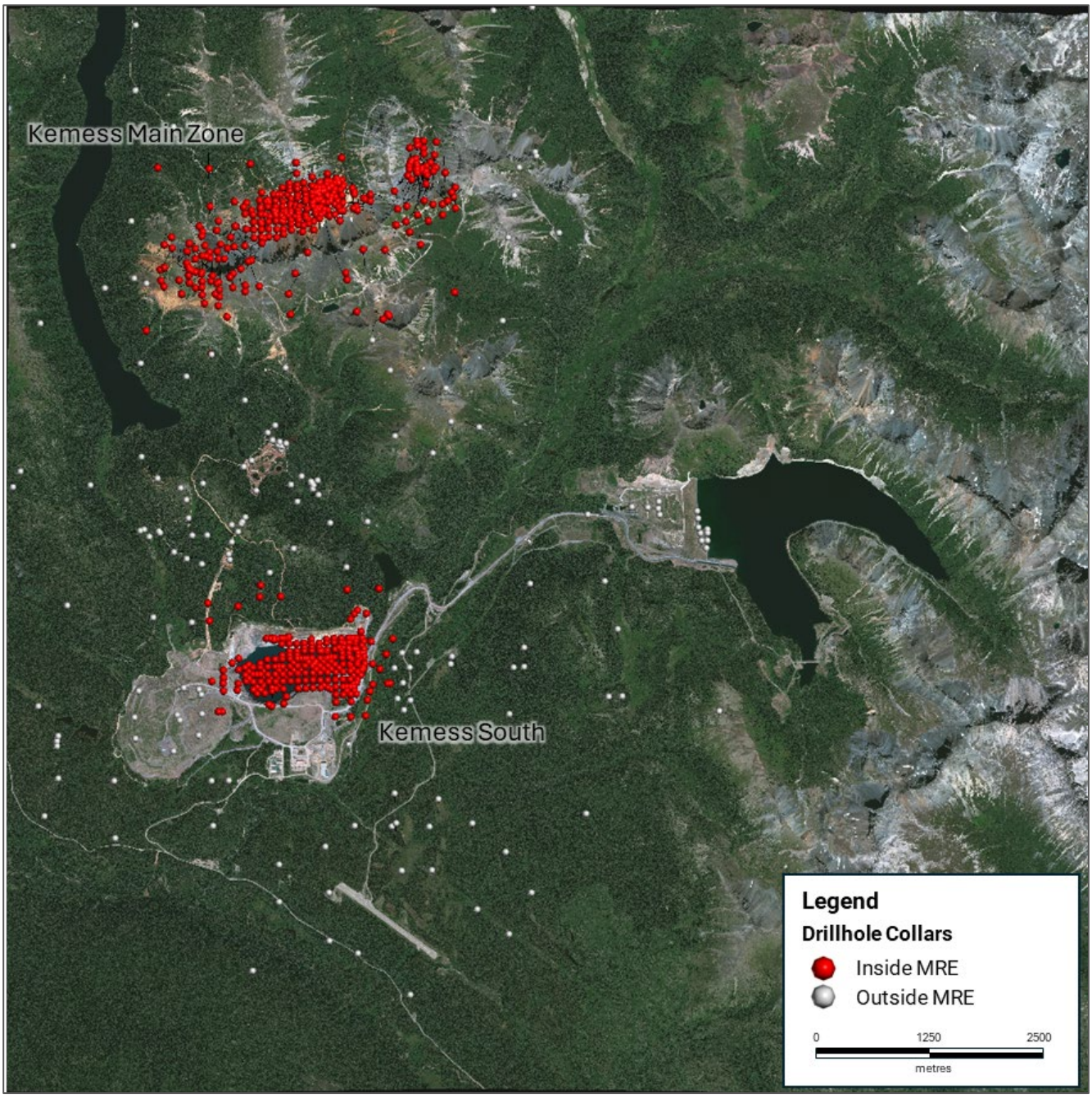
The effective date of the Kemess South drilling database is November 18, 2025. The 2025 Kemess South MRE utilises 273 drillholes (46,309 m), excluding 634 drillholes (291,995 m) as they are outside of the main mineralized domains.

Routine validation of the drilling database is performed to maintain data accuracy and consistency. Historical drillhole and assay data used in the MRE have been reviewed for completeness, accuracy, and consistency and potential biases were investigated. The data were deemed appropriate for inclusion in the MRE. Gaps in the analytical quality control data at Kemess South discussed in Item 11 were considered; however, a statistical comparison between data that are supported by industry standard analytical quality control data and those collected prior to 2003 without such support suggest that the

latter data are not biased and thus were deemed appropriate for the inclusion in this study. The classification of mineral resources at Kemess South considered the above.

Figure 14-1 shows the drillholes considered for the MRE.

Figure 14-1: Kemess property drillhole collar locations



14.4 GEOLOGICAL INTERPRETATION AND MODELLING

14.4.1 Kemess Main Zone

Litho-structural, alteration, and rock quality models were created by Centerra in 2024 for Kemess Main Zone. These models include Kemess Main, Nugget, KUG deposit, and the Kemess East deposits and incorporates interpretations from previous models (specifically fault modelling), drill hole data from historic and current drill programs, and interpretations from the lithogeochemical alteration model conducted by CSA Global in 2020 and updated in 2024 (detailed in Item 7.4). Drill hole data used in the model include:

- Detailed drill hole logging
- Whole rock geochemical assay data
- Aqua Regia geochemical assay data
- Magnetic susceptibility data
- AISIRUS Terraspec data
- RQD.

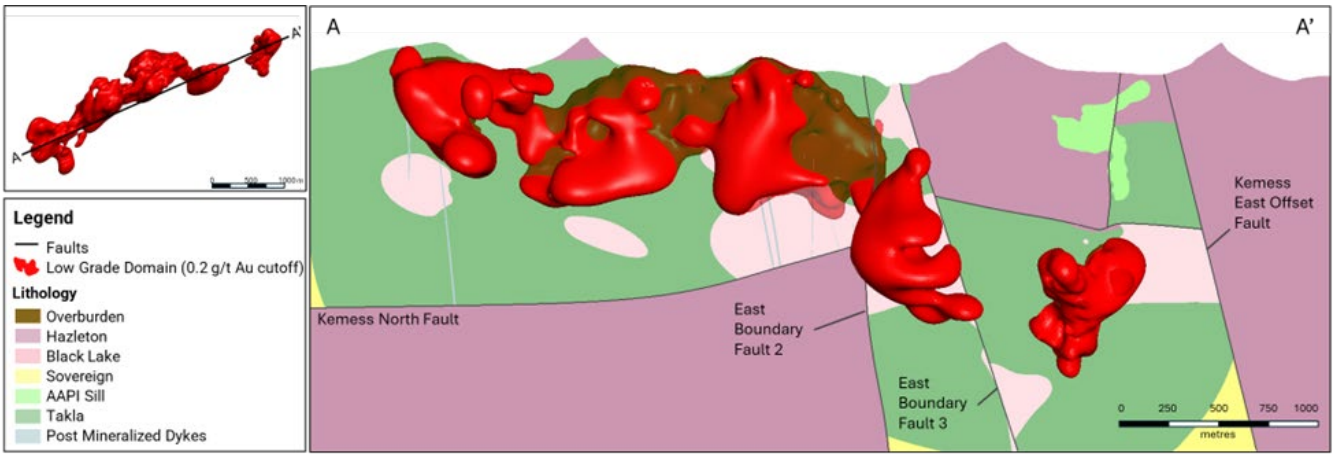
Lithology

The geological model is based primarily on information collected during core logging. Detailed geological information was combined into seven grouped lithologies, listed in order of emplacement in Table 14-1. Figure 14-2 shows a cross-section view of the modelled geology.

Table 14-1: Modelled lithology units

Lithology	Description
Takla	Undifferentiated Takla Group volcanic rocks
AAP Sill	Augite Andesite Phyric rocks, part of the Takla Group
Black Lake	Intrusive rocks of the Black Lake Suite
Post Mineral Dykes	Intrusive rocks emplaced after the mineralizing event
Sovereign	Sovereign Pluton
Hazelton	Volcanic breccias of the Hazelton Group
Overburden	

Figure 14-2: Litho-structural model for Kemess deposits shown in long-section view, looking north-northwest



Faults

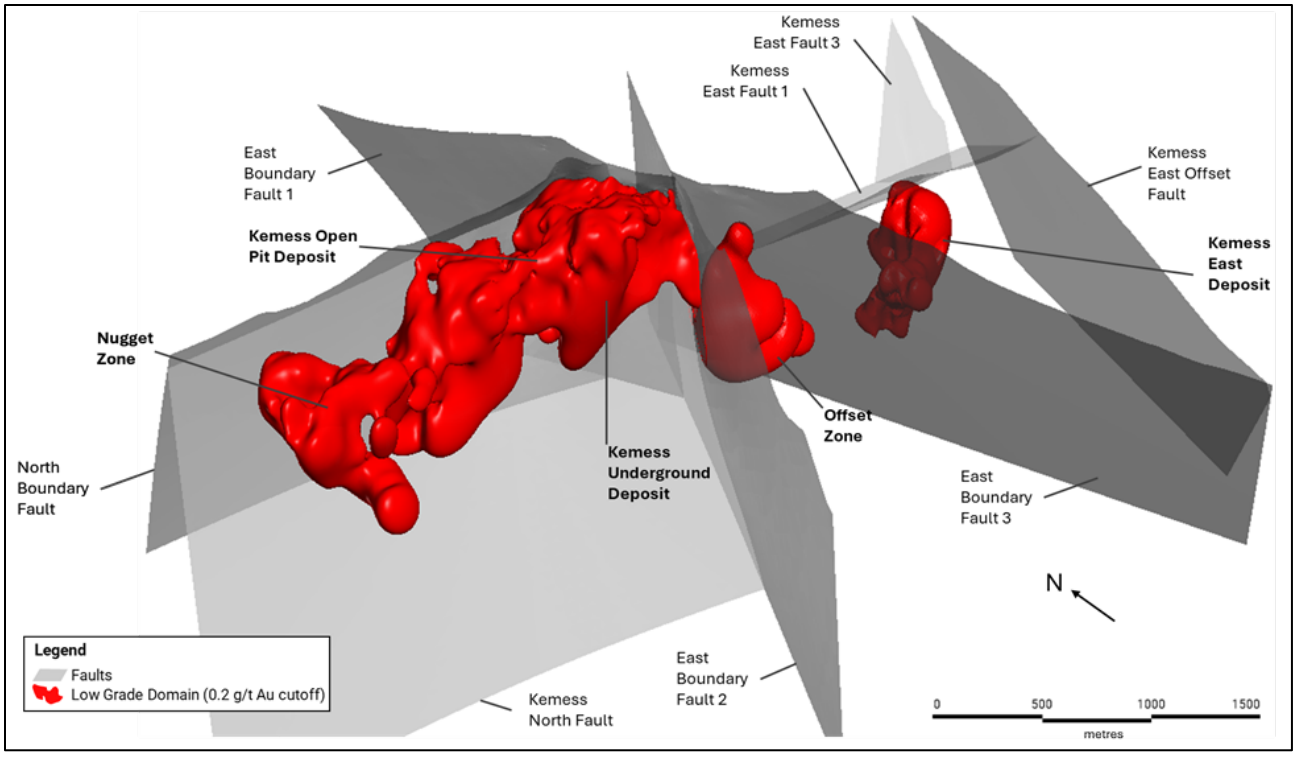
The Kemess Main Zone deposits have undergone extensive post-mineral faulting, evident in cliff-wall exposures and drill core. These observations informed the development of a fault model, currently represented by planes offsetting lithological contacts and grade domains (Figure 14-3). Table 14-2 outlines the temporal relationships used in the model development. The current fault model utilizes eight of the nine modelled faults, dissecting the lithological model to create 10 individual fault blocks.

Table 14-2: List of major modelled faults

Sequence	Fault	Kinematics/Orientation	Relationships
Oldest	Kemess North Fault	South dipping thrust fault, truncating KUG ore zone to north and at depth	Truncates against East Boundary Fault 2.
	North Boundary Fault	North dipping normal fault, truncating KUG ore zone to north	Truncates against East Boundary Fault 2.
	KE Fault 3	West dipping normal fault	Truncates against East Boundary Fault 2. Truncates against Kemess East Offset Fault. Truncates against KE Fault 1.
	KE Fault 1	North dipping normal fault	Truncates against East Boundary Fault 2. Truncates against Kemess East Offset Fault.
	East Boundary Fault 1/3	Northeast dipping normal fault, truncating KUG ore zone to northeast	Cut by East Boundary Fault 2.
	East Boundary Fault 2	East dipping normal fault, truncating KUG ore zone to north	
Youngest	Kemess East Offset Fault	Truncates Kemess East ore zone to east	

The East Boundary Fault 1/3 is cut and offset by East Boundary Fault 2. For modelling purposes, the Northern segment is referred to as East Boundary Fault 1 and the southern segment is referred to as East Boundary Fault 3.

Figure 14-3: Kemess Main Zone Trend fault model



Note: Viewed obliquely from above towards the NE. Significant faults, deposits and zone are labelled.

Alteration Model

The alteration model was built with a very similar workflow to the lithology model, involving interval selections based on primary drilling data for logged alteration intervals. Selections were validated using magnetic susceptibility, whole rock assay and spectral data.

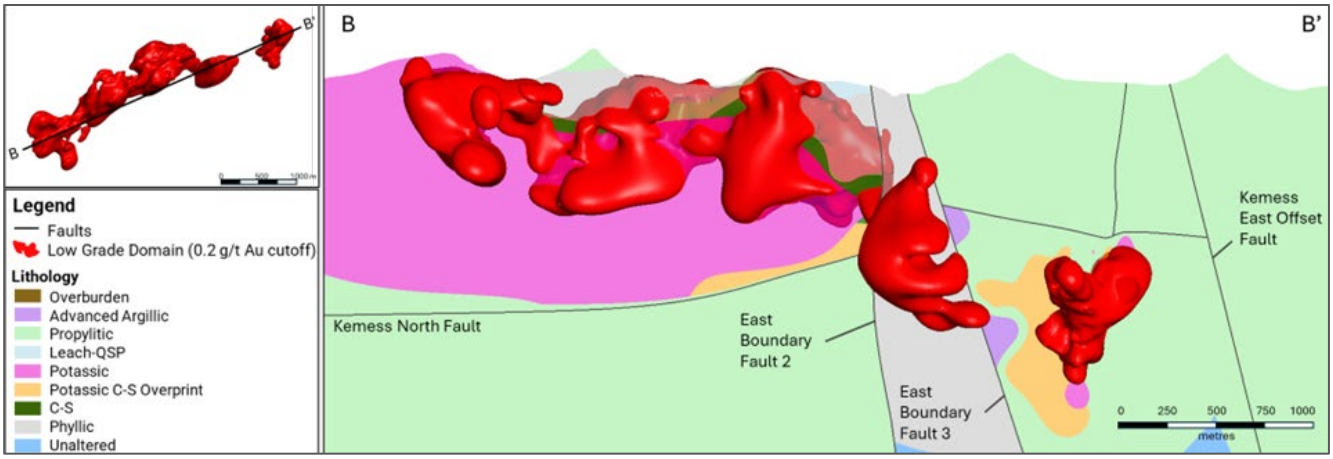
CSA Global was contracted by Centerra in December 2024 to analyze the lithogeochemical data and create an alteration model. The alteration model was validated against the CSA Global lithogeochemical model. Minor edits were made to domain boundaries with insights gained from the CSA Global model.

The alteration model consists of nine domains, detailed in Table 14-3. A long section view of the alteration model is shown in Figure 14-4.

Table 14-3: Modelled alteration assemblages

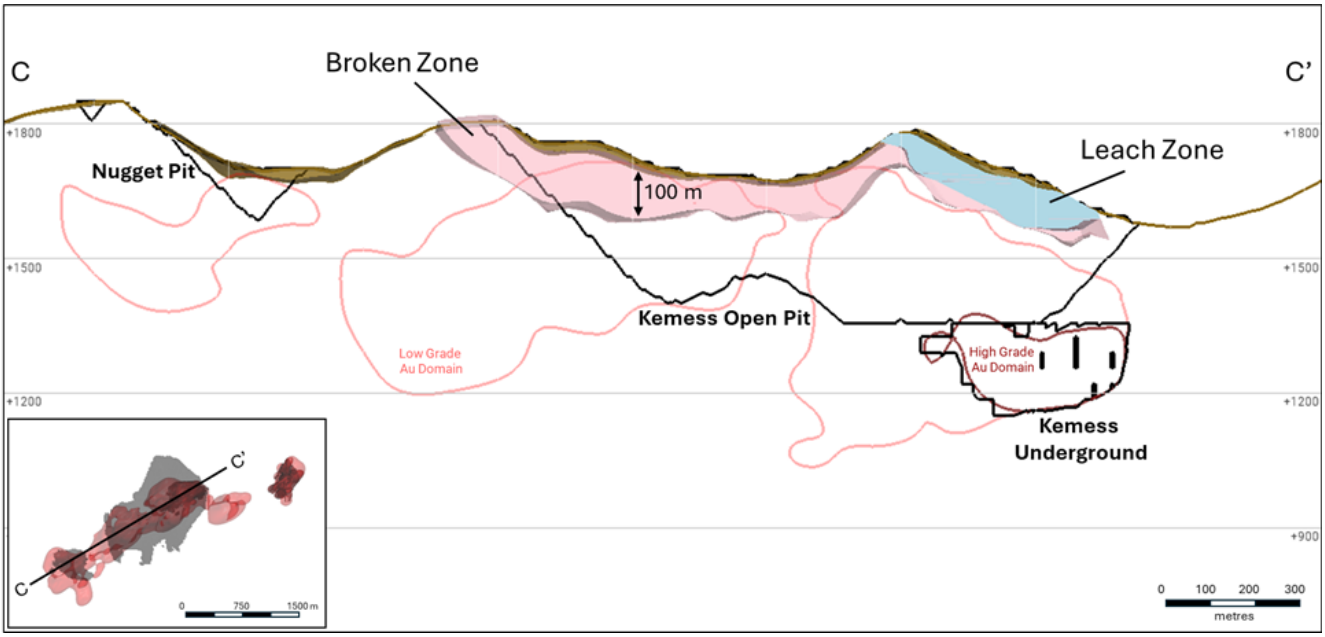
Code	Description
Potassic	Alteration minerals dominated by magnetite ± biotite ± k-feldspar
Potassic C-S Overprint	Alteration minerals dominated by magnetite ± biotite with a chlorite ± sericite overprint
C-S	Alteration minerals dominated by chlorite and sericite
Phyllic	Alteration minerals dominated by quartz, sericite ± pyrite
Advanced Argillic	Alteration minerals dominated by clay
Leach QSP	Alteration minerals dominated quartz, sericite, pyrite and clay
Propylitic	Alteration minerals dominated by epidote and chlorite
Unaltered	

Figure 14-4: Alteration model for Kemess Main Zone Trend deposits (long-section view, looking north-northwest)



In addition to lithology and alteration domains, a rock quality domain was defined in a separate model. This zone of poor rock-quality is commonly known as the “Broken Zone” and shown in longitudinal section in Figure 14-5. This domain is not typified by a specific lithological texture or composition, or by alteration assemblage, but is clearly defined by low RQD values and calcium depletion, calcium values ranging from 0.1% to 3.0% Ca. Background Ca ranges from 3% to 12% Ca. The Broken Zone is a result of hydration of anhydrite veins to gypsum, followed by dissolution of the gypsum veins by ground waters. The hydration of anhydrite to gypsum corresponds to a 60% volume expansion, resulting in fracturing of the rock mass.

Figure 14-5: Long section of the Kemess deposits showing leach and broken zones, looking north-northwest



The Leach QSP zone, which is an alteration domain, is typified by strong quartz-sericite-pyrite alteration, poor rock competence, and intense calcium depletion; calcium values ranging from 0.10% to 0.05% Ca. This zone is contained within the Broken Zone and is the product of surface weathering of pyrite, which

generates acidic groundwater that causes variable clay alteration. Weathering of pyrite is encouraged by the high permeability of the Broken Zone.

14.4.2 Kemess South

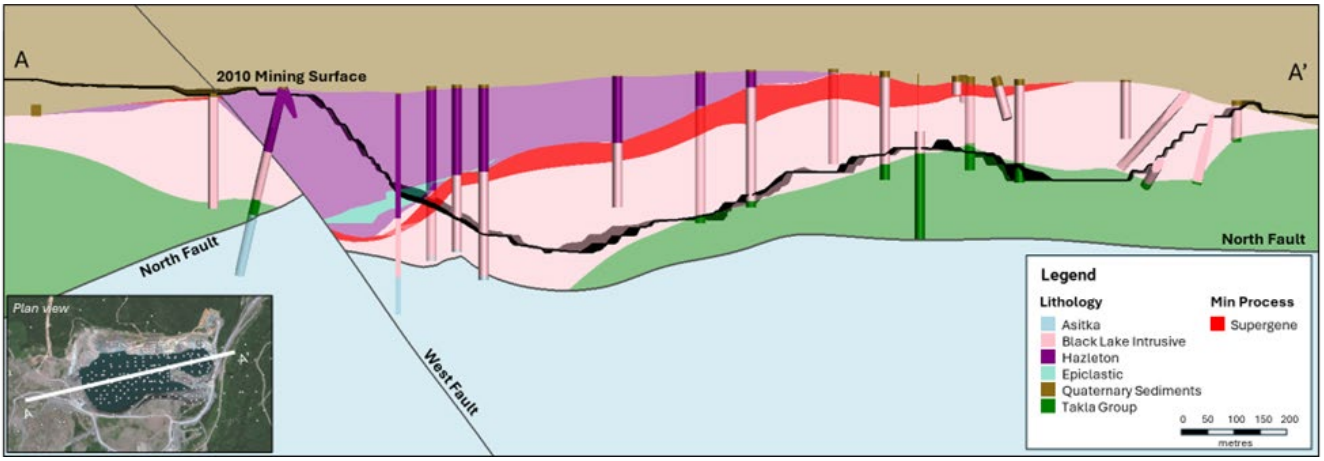
Lithology

A historical litho-structural model from 2008 was updated to incorporate more recent drillholes. The geological model is based primarily on information collected during core logging. The epiclastic rocks were based off the 2008 modelled solid. Detailed geological information was combined into seven grouped lithologies, listed in order of emplacement in Table 14-4. Figure 14-6 shows a cross-section view of the modelled geology.

Table 14-4: Modelled lithology units

Lithology	Description
Asitka	Siltstone and limestones of the Asitka Group
Takla	Undifferentiated Takla Group volcanic rocks
Black Lake	Intrusive rocks of the Black Lake Suite
Hazelton	Volcanic breccias of the Hazelton Group
Epiclastic	Epiclastic rocks of the Toodoggone Formation of the lower Hazelton Group
Overburden	

Figure 14-6: Kemess South litho-structural and supergene enrichment models (cross-section view)



Supergene Enrichment

A blanket of copper-enriched supergene mineralization overlies hypogene mineralization and comprises approximately 20% of the deposit. Most of the supergene mineralization has been mined. This enrichment was modelled separately and is shown overlain on the lithological model in Figure 14-6.

Faults

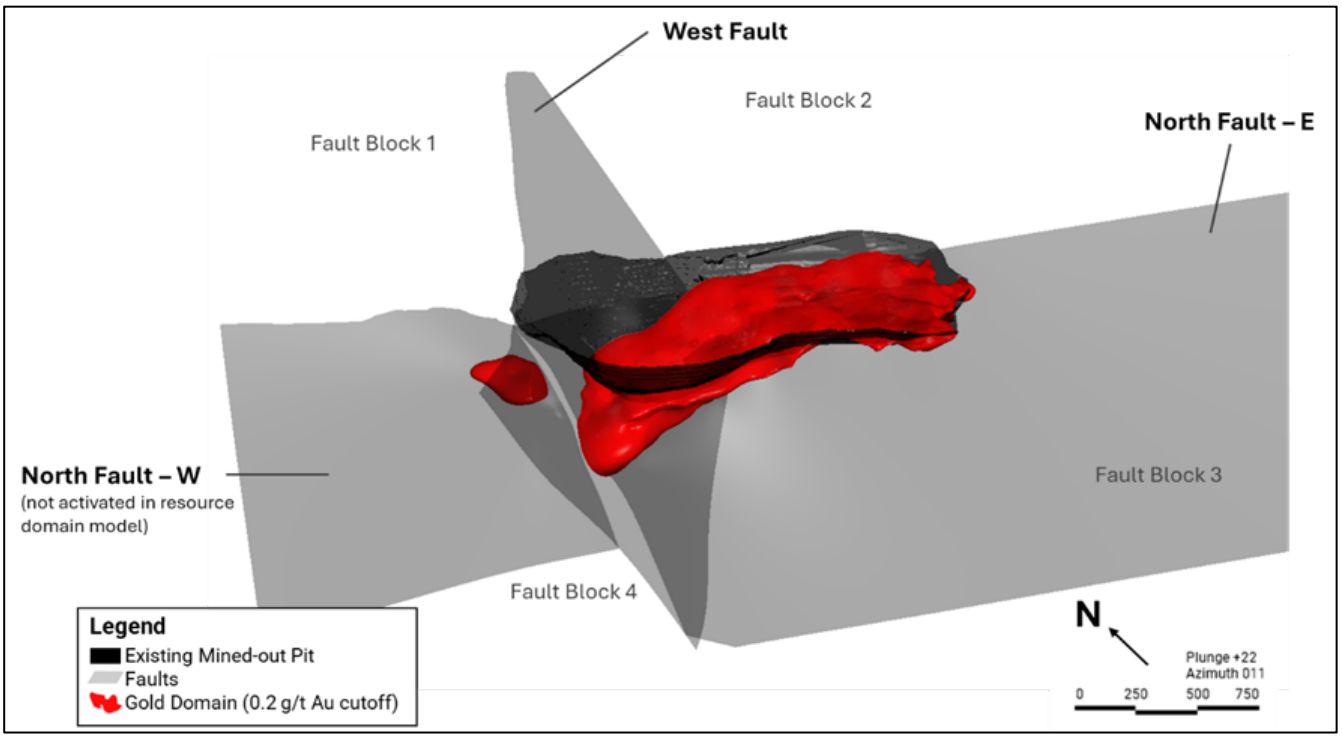
The Kemess South deposit has undergone post-mineral faulting, evident in pit-wall exposures and drill core. These observations informed the development of a fault model, currently represented by planes

offsetting lithological contacts and grade domains. Table 14-5 outlines the temporal relationships used in its development. The current fault model utilizes three modelled faults, dissecting the lithological model to create four individual fault blocks (Figure 14-7).

Table 14-5: List of major modelled faults at Kemess South

Fault plane (youngest to oldest)	Interaction type	Following plane	Side
West Fault			
North Fault E	Terminates against	West Fault	West
North Fault W	Terminates against	West Fault	East

Figure 14-7: Kemess South fault model, viewed obliquely from above towards the northeast



14.5 MINERALIZATION CONTROLS AND GRADE DOMAIN MODELLING

14.5.1 Kemess Main Zone

In 2025, Centerra updated the resource estimates at Kemess Main Zone based on new drilling and understanding of the deposit. The 2025 MRE includes both open pit and underground resources. The model incorporates gold, copper, and silver grade domains at Kemess Main Zone and Kemess South and considers alteration zones for its capping strategy at Kemess Main Zone.

Interpretation of the mineralization of the Kemess deposits is supported by surface drilling. Grade domains were modeled primarily from assay trends, guided by geological understanding. Logged data including lithological boundaries, alteration, and structural controls were used to ensure spatially

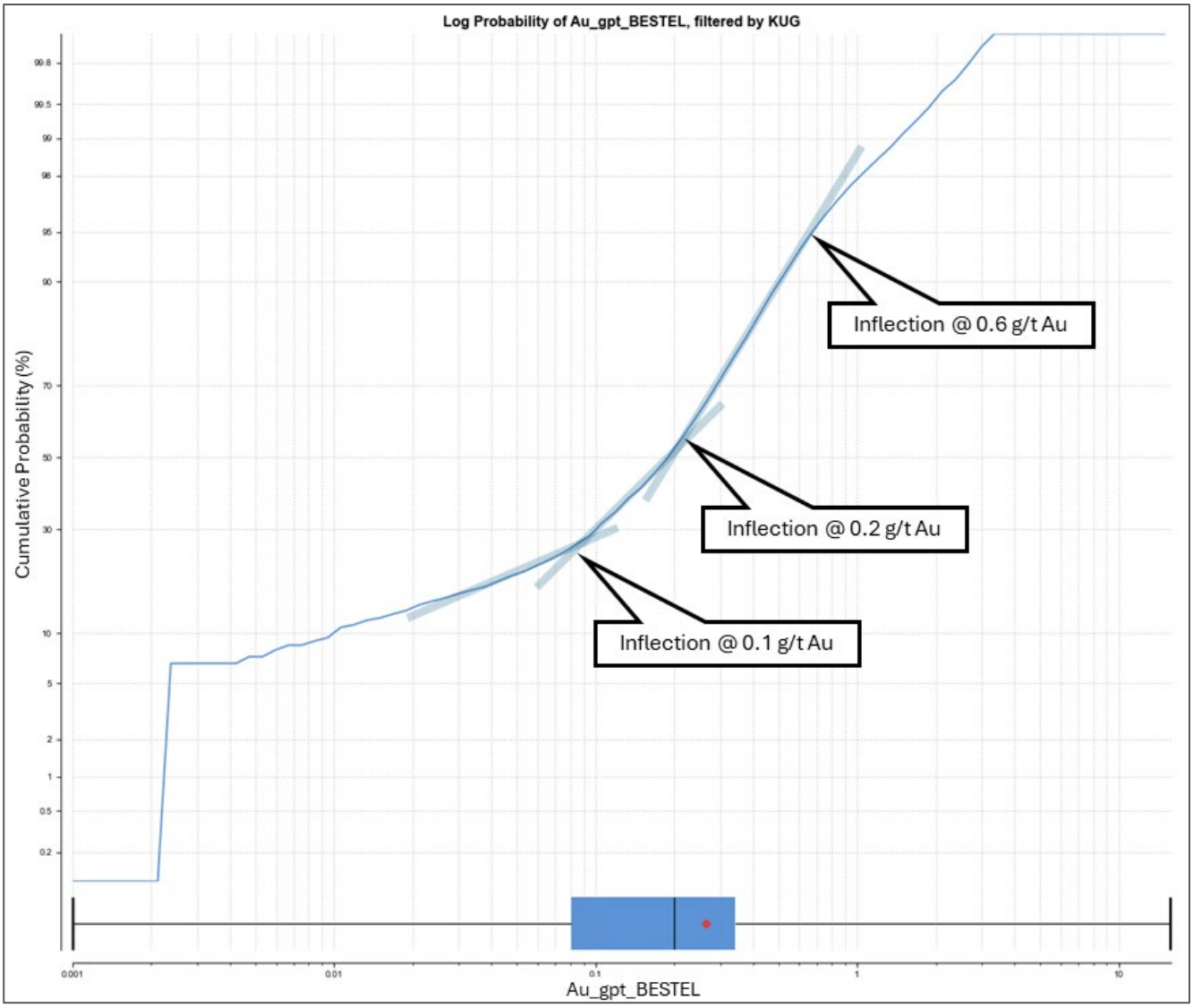
consistency. The modeled grade domains were used to estimate block grades and to prevent grade smearing and dilution due to mixing of statistical populations.

Probability plots of all assays were evaluated to examine the gold, copper, and silver distribution and identify a potential threshold value to discriminate for mineralization. Figure 14-8 and Figure 14-9 show examples of these plots for gold assay analysis. The same methodology was applied for copper and silver assays. At Kemess Main Zone, the data indicate 0.2 g/t Au and 0.6 g/t Au as threshold values to differentiate the low-grade and high-grade gold mineralization envelopes respectively from surrounding host rock in the KUG deposit and 0.3 g/t Au and 0.6 g/t Au for the low-grade and high-grade mineralization envelopes respectively in the Kemess East deposit. The low-grade threshold from the KUG data review was implemented for the Nugget and Offset deposits as they are considered continuous to the KUG deposit. Copper and silver mineralization at the Kemess Main Zone is not present in the Nugget deposit. Silver mineralization at the Kemess Main Zone does not support continuous high-grade domains; thus, only low-grade silver mineralization envelopes were constructed. Table 14-6 summarizes threshold values for domain modelling for gold, copper, and silver mineralized domains within the Kemess Main Zone project.

Table 14-6: Kemess Main Zone grade domain modelling criteria

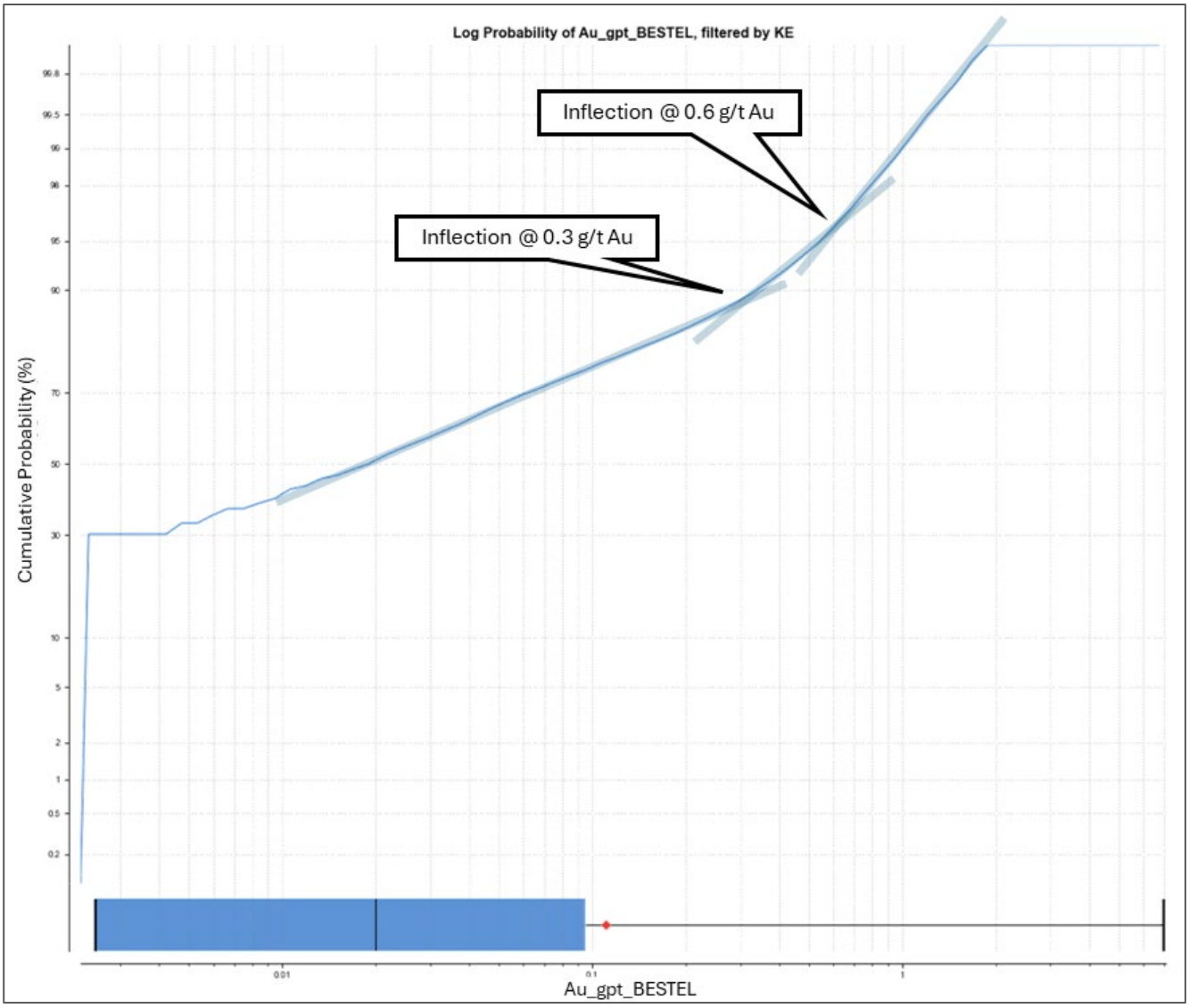
Project	Mineralized domain	Grade criteria
Kemess Main Zone	Au – Low grade	Nugget, KNOP: 0.2 g/t Au KE: 0.3 g/t Au
	Au – High grade	KUG, KE: 0.6 g/t Au
	Cu – Low grade	Nugget, KNOP, KE: 0.2% Cu
	Cu – High grade	KE: 0.4% Cu
	Ag – Low grade	KUG: 1.5 g/t Ag KE: 2 g/t Ag

Figure 14-8: Log probability plot of all gold assays in the KUG area



Note: Plot indicates population breaks at 0.2 g/t Au and 0.6 g/t Au.

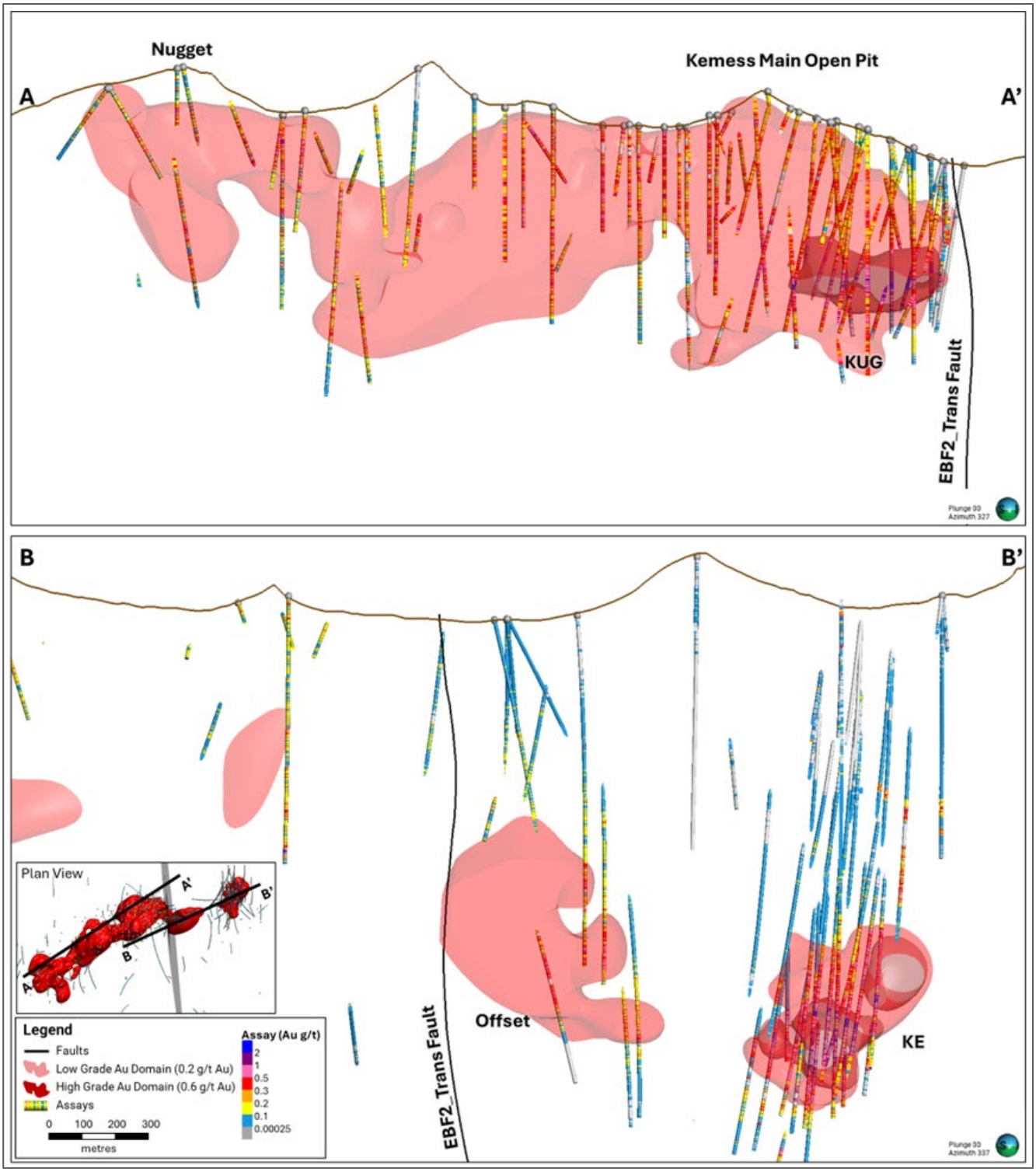
Figure 14-9: Log probability plot of all gold assays in the Kemess East area



Note: Plot indicates population breaks at 0.3 g/t Au and 0.6 g/t Au.

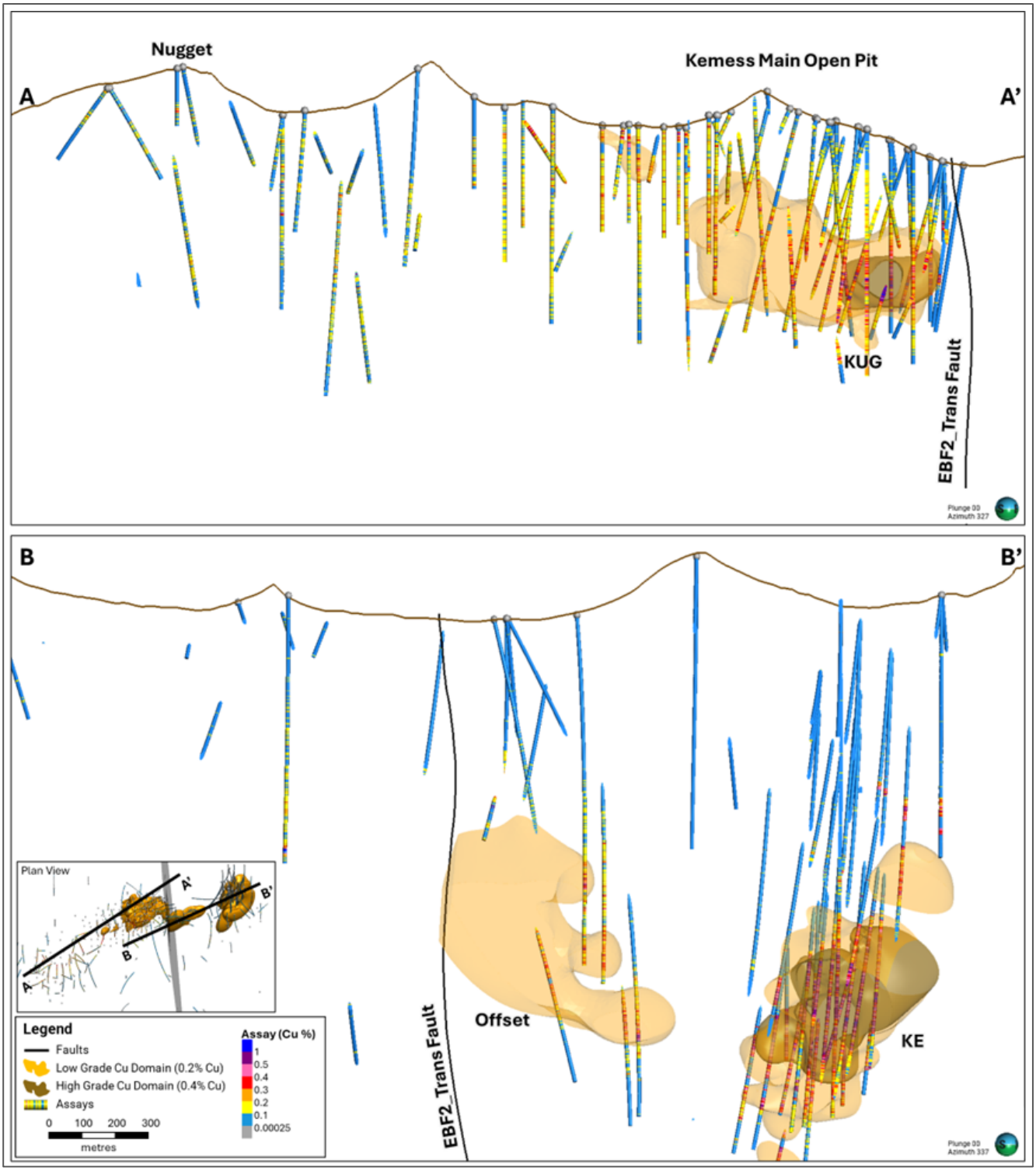
The orientations of the modelled mineralized domains follow the Black Lake intrusives, striking northwest and dipping 60° southeast. The EBF2 fault structurally offsets the low-grade domain creating separate structural domains and mineralization volumes (Figure 14-10 to Figure 14-12). Material outside the defined grade domains were considered background mineralization. These areas were informed by assay data external to the grade envelopes ensuring appropriate grade distribution and minimizing artificial grade smearing across grade and structural boundaries.

Figure 14-10: Modelled gold domains at Kemess Main Zone



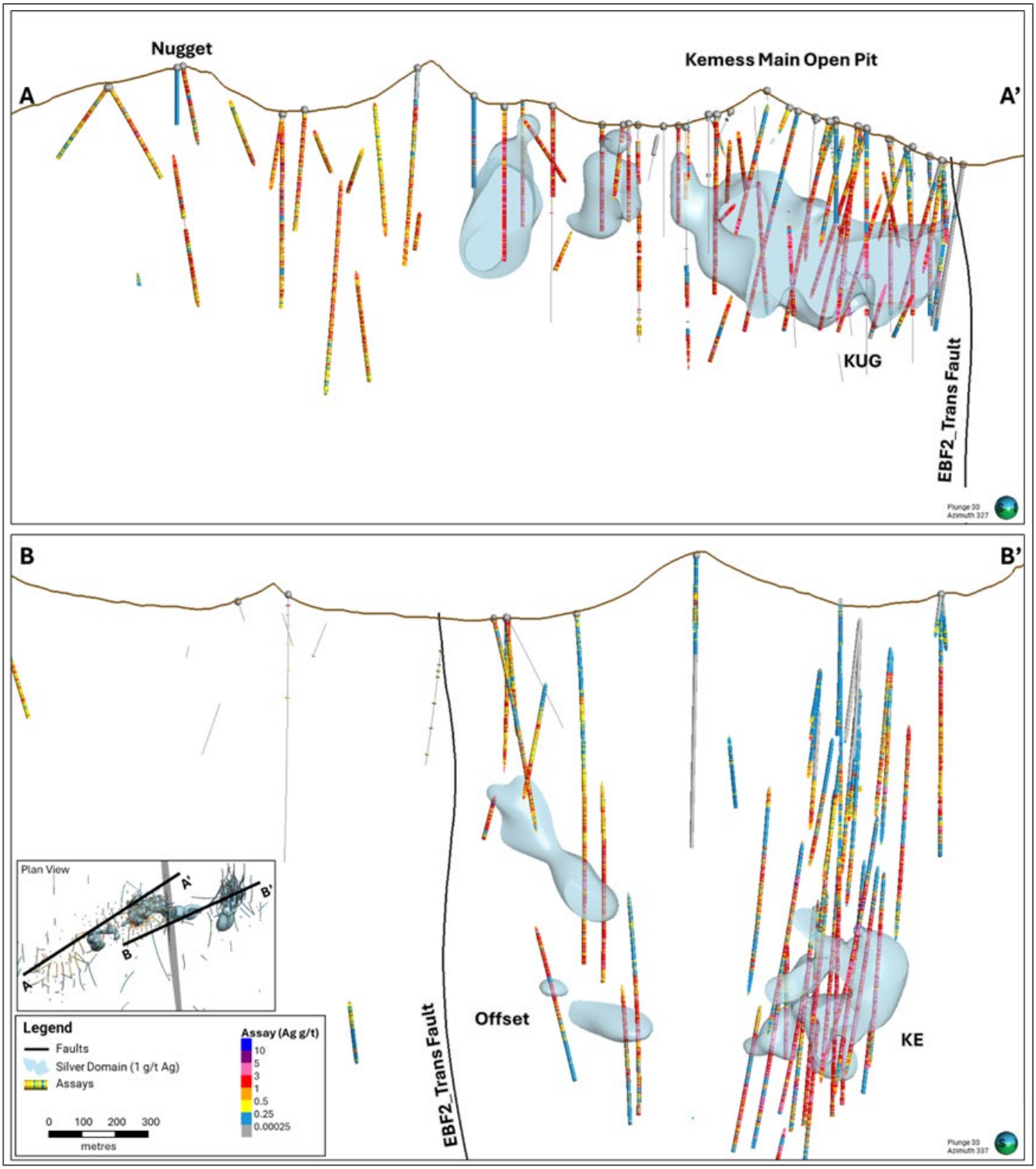
Note: A-A' section: Nugget, Kemess Main open pit, and KUG mineralized domains; B-B' section: Offset and KE mineralized domains.

Figure 14-11: Long-section views of the modelled copper domains at Kemess



Note: A-A' section: Kemess main open pit and KUG mineralized domains; B-B' section: Offset and KE mineralized domains.

Figure 14-12: Long-section view of the modelled silver domains at Kemess



Note: A-A' section: Kemess Main open pit and KUG mineralized domains; B-B' section: Offset and KE mineralized domains.

14.5.2 Kemess South

The current Kemess South MRE supersedes the historical 2007 estimate and incorporates additional drilling completed in 2008, 2010, 2019. Similar to the Kemess Main Zone model, Kemess South incorporates gold, copper, and silver grade domains.

Interpretation and modelling of the mineralization of the Kemess South deposits follows similar procedures used at Kemess Main Zone, utilizing surface drilling and logged data. The modeled grade domains were used to estimate block grades and to prevent grade smearing and dilution due to mixing of statistical populations.

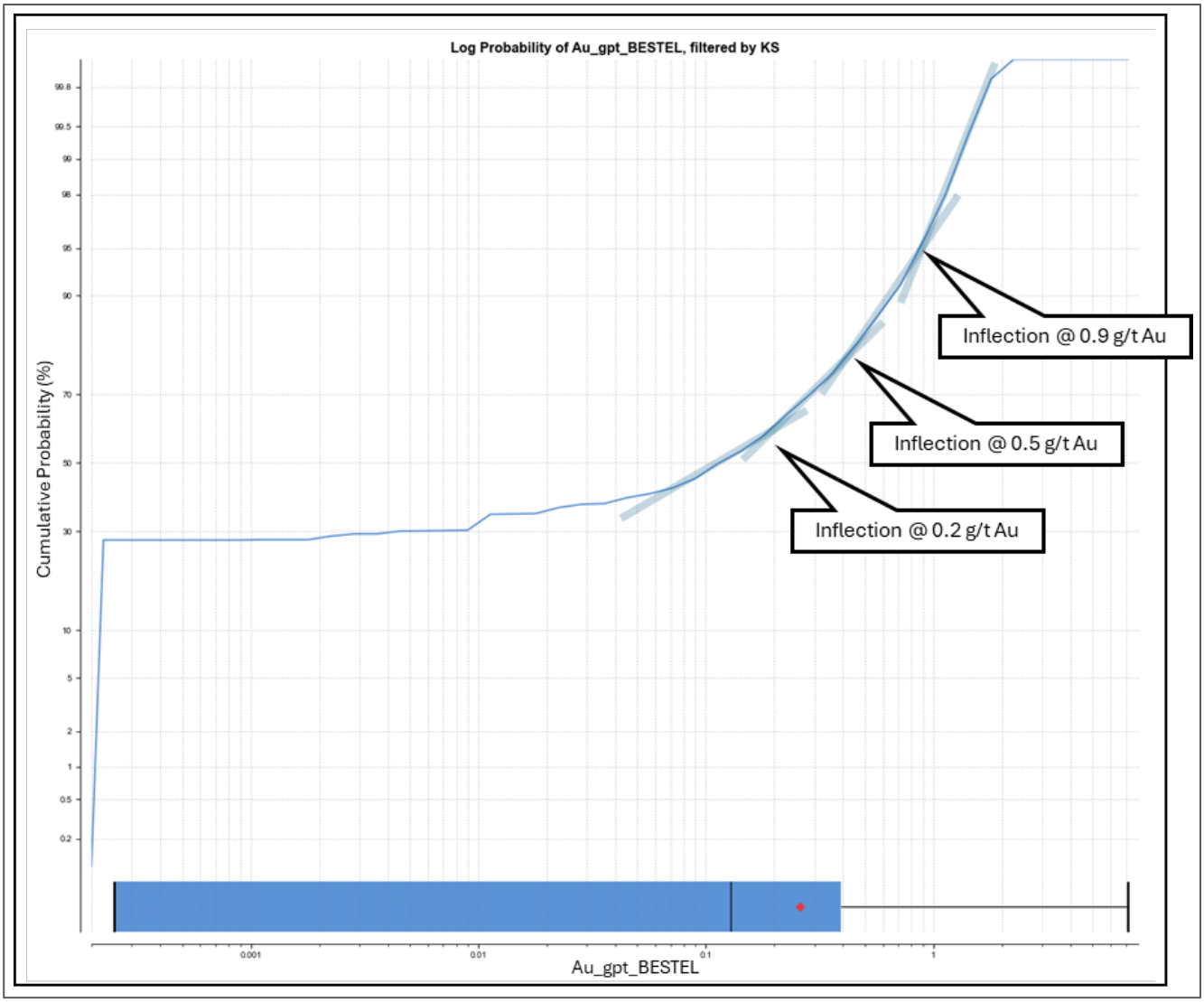
Probability plots of all raw assays were evaluated to examine the gold, copper, and silver distributions and identify potential threshold values to discriminate for mineralization (Figure 14-13). Table 14-7 summarizes threshold values for domain modelling for all mineralized domains at Kemess South.

Using the intrusion tool in the Leapfrog Geo™ software, low-grade and high-grade volumes were modelled from mineralized economic composite intervals that meet various grade and length criteria. The orientations of these volumes follow the Black Lake intrusives, dipping 15 degrees to the southwest. The West fault structurally offsets the mineralized domain creating separate structural domains and mineralization volumes (Figure 14-14, Figure 14-15 and Figure 14-16). Material outside the defined grade domains were considered background mineralization. These areas were informed by assay data external to the grade envelopes ensuring appropriate grade distribution and minimizing artificial grade smearing across grade and structural boundaries.

Table 14-7: Kemess South grade domain modelling criteria

Project	Mineralized domain	Grade criteria
Kemess South	Au	0.2 g/t Au
	Cu	0.2% Cu
	Ag	1 g/t Ag

Figure 14-13: Log probability plot of all gold assays in Kemess South



Note: Plot indicates population breaks at multiple grades; 0.2 g/t Au was chosen for low-grade domain modelling.

Figure 14-14: Oblique view of Kemess South mineralized gold domains

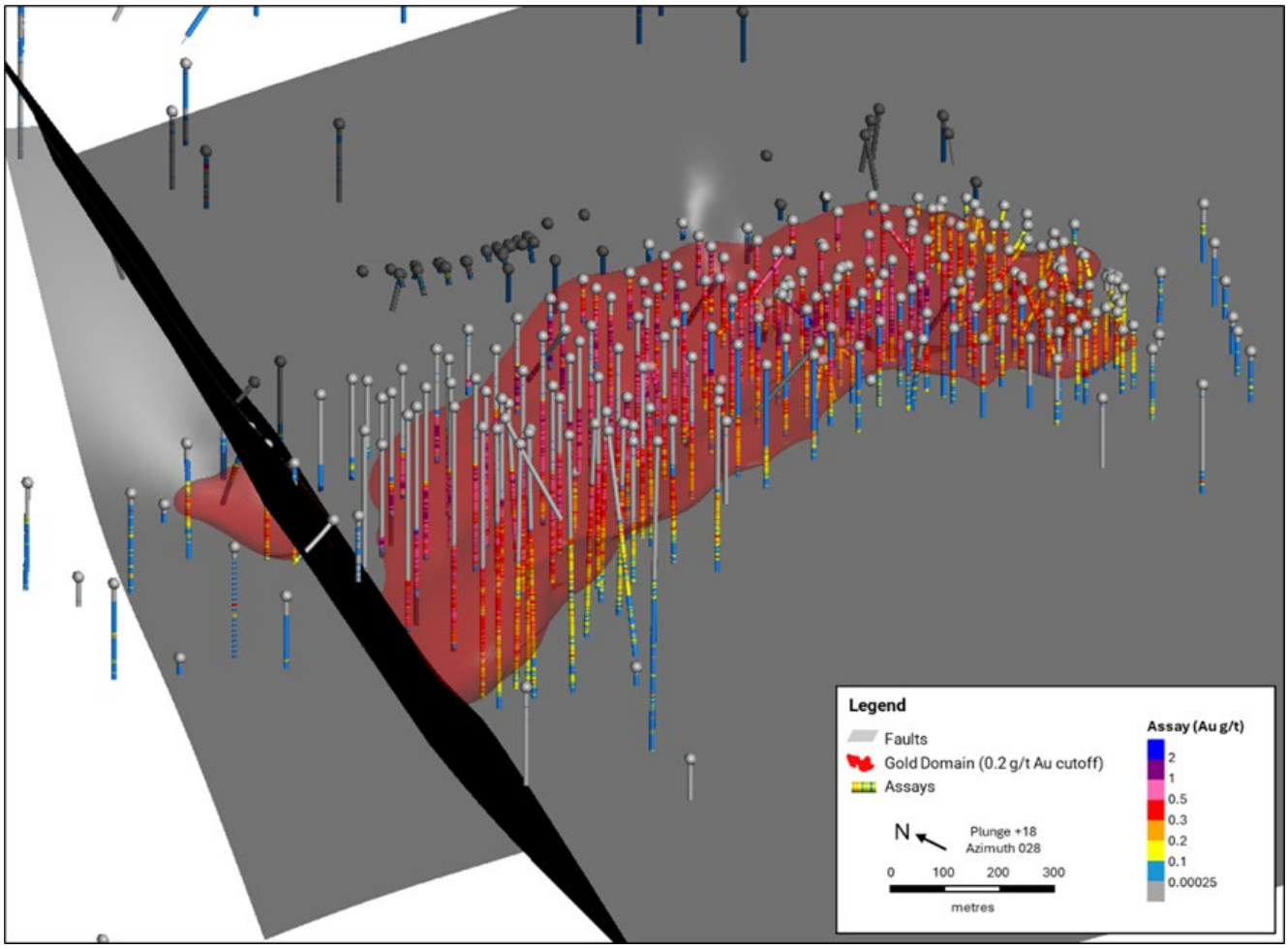


Figure 14-15: Oblique view of the Kemess South mineralized copper domains

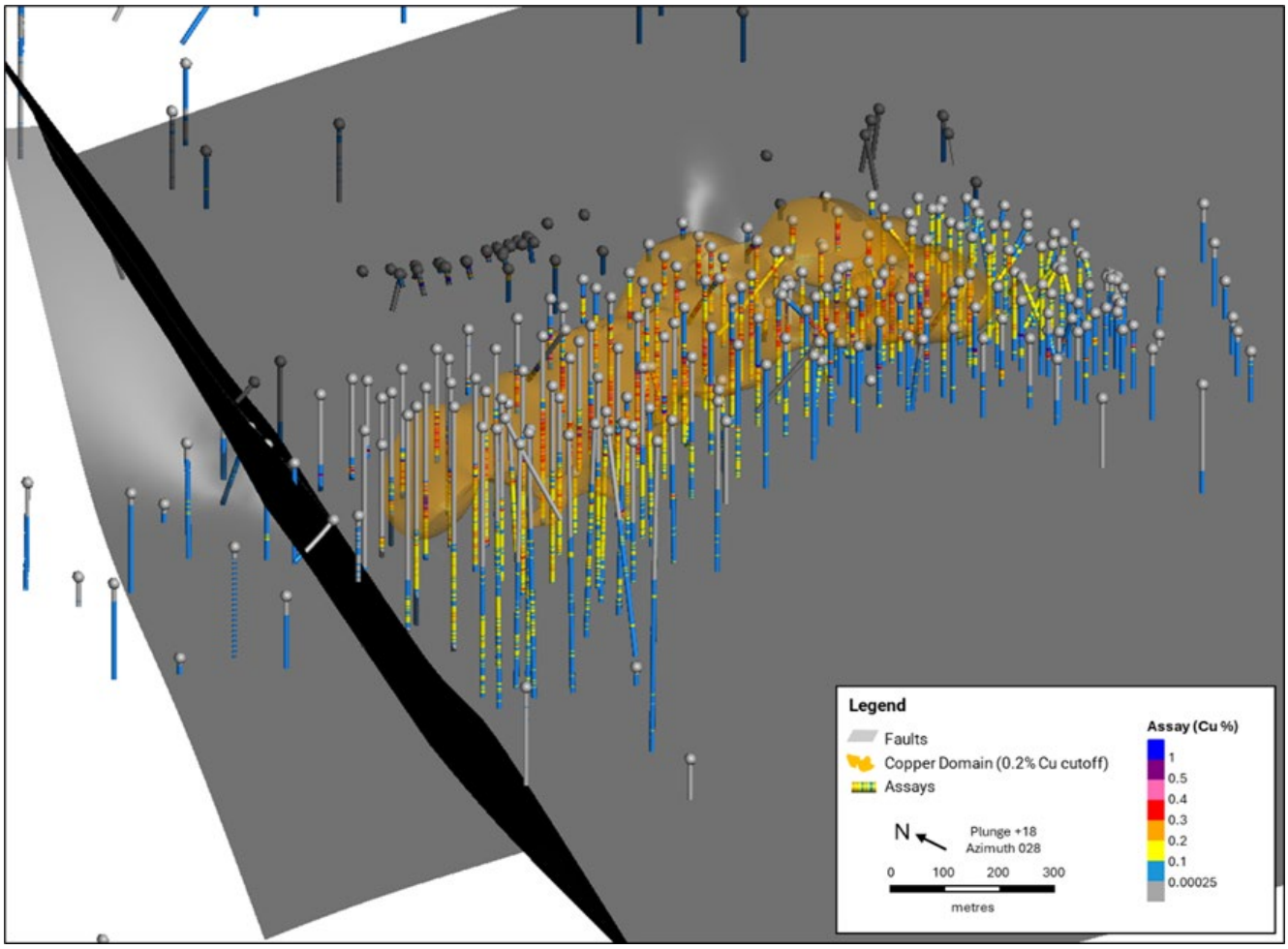
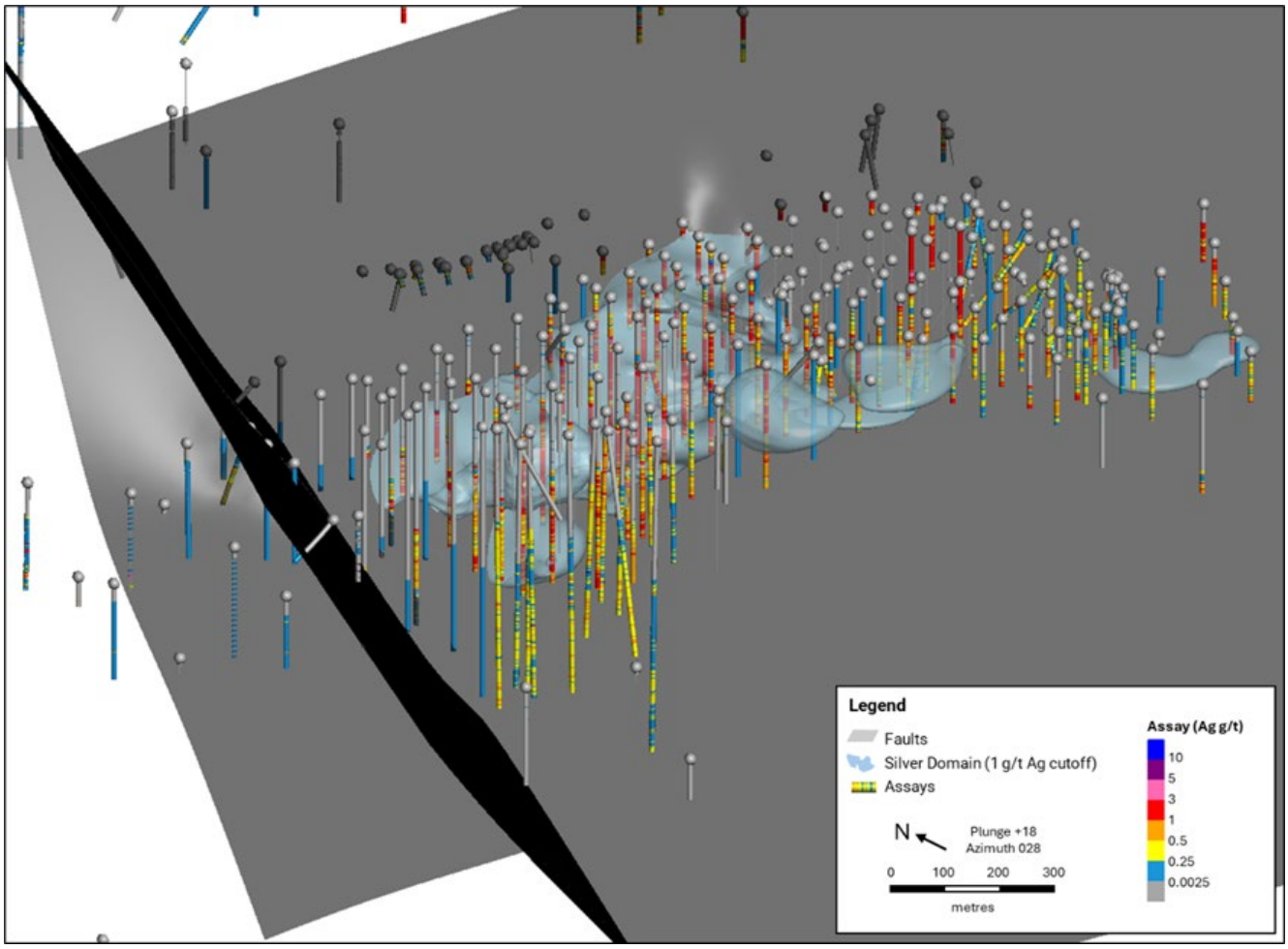


Figure 14-16: Oblique view of the Kemess south mineralized silver domains



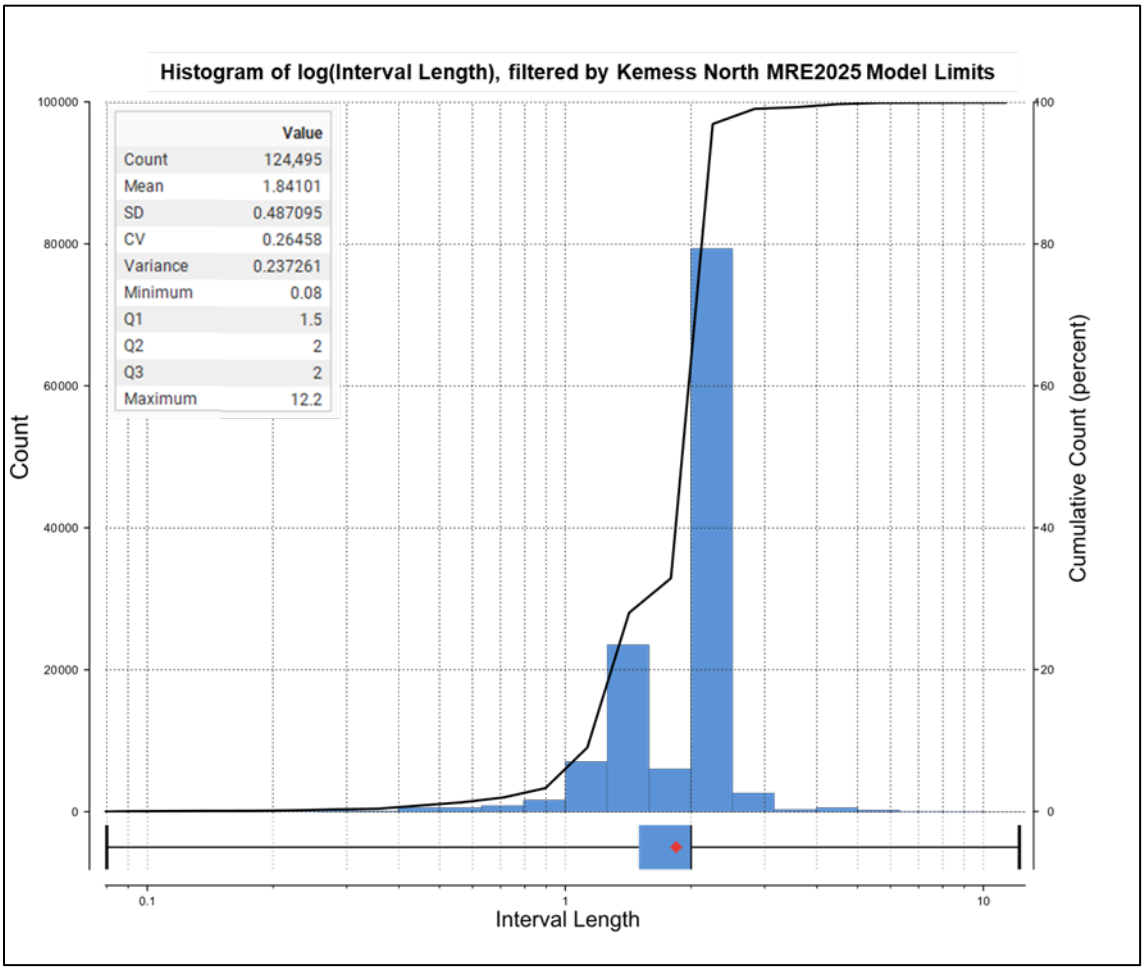
14.6 COMPOSITING SAMPLES AND CAPPING ASSAYS

14.6.1 Kemess Main Zone

A review of the distribution of assay lengths in the database showed that over 90% of assays were sampled at 2 m intervals or less in the Kemess Main Zone deposit. Assay data were composited to 2 m composites, with a minimum sample coverage of 1 m. Residual end lengths shorter than 1 m were distributed equally throughout the drillhole. This choice of compositing length considered the potential block sizes to be estimated (15 m x 15 m x 15 m for open pit scenarios and 5 m x 5 m x 5 m for those portions of the deposit considered for underground mining scenarios. Block sizes considered general drill hole spacings, common bench height in open pit scenarios, and a minimum mining width for underground scenarios). Compositing was performed within the boundaries of mineralization domains but were not split at the alteration or lithological boundaries. This approach assumes that all lithological and alteration domains have soft boundaries for the purpose of Mineral Resource estimation consistent with broad mineralization and alteration patterns typical in porphyry deposits.

Figure 14-17 shows the distribution of assay lengths of the Kemess Main Zone assay data. Approximately 94% of assays within the resource model area are less than 2 m in length.

Figure 14-17: Sample length distribution and associated statistics



A summary of length-weighted gold, copper, and silver assays within the mineralized domains can be found in Table 14-8, Table 14-9, and Table 14-10, respectively.

Table 14-8: Summary statistics of length-weighted gold assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	All	124,474	0.17	0.28	1.59	0.00	0.01	0.08	0.24	17.70
	KE_0.6	24,864	0.35	0.24	0.69	0.00	0.22	0.31	0.43	17.70
	KE_0.3	1,701	1.18	0.74	0.63	0.00	0.70	0.96	1.42	6.12
	KUG_0.6	3,232	0.46	0.20	0.44	0.00	0.34	0.44	0.56	2.41
	Nugget_KUG_0.2	1,774	0.85	0.36	0.43	0.00	0.62	0.77	1.02	3.85
	Offset_0.2	1,169	0.29	0.21	0.70	0.00	0.19	0.26	0.36	6.50
	Unknown	91,734	0.08	0.16	1.97	0.00	0.01	0.04	0.11	15.75

Table 14-9: Summary statistics of length-weighted copper assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	All	124,495	0.10	0.14	1.42	0.00	0.01	0.04	0.14	5.17
	Nugget_KUG_0.2	7,377	0.27	0.15	0.55	0.00	0.19	0.25	0.32	5.17
	KUG_0.4	797	0.55	0.26	0.47	0.01	0.39	0.50	0.66	4.15
	KE_0.2	4,752	0.34	0.16	0.46	0.00	0.24	0.32	0.42	1.99
	KE_0.4	3,105	0.51	0.17	0.33	0.00	0.41	0.49	0.60	2.57
	Offset_0.2	972	0.27	0.16	0.61	0.00	0.19	0.25	0.33	3.04
	Unknown	107,492	0.06	0.08	1.34	0.00	0.01	0.03	0.09	4.15

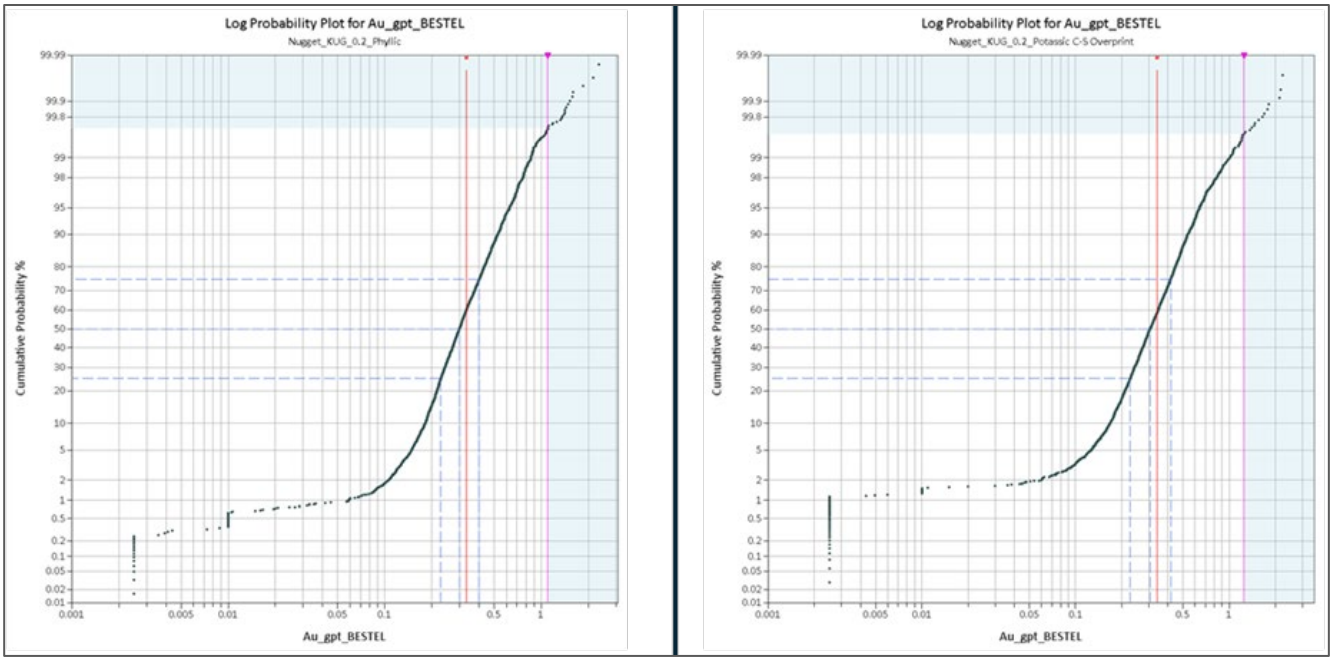
Table 14-10: Summary statistics of length-weighted silver assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	All	106,828	0.91	1.32	1.44	0.00	0.25	0.51	1.20	104.00
	KUG_1.5	8,105	2.23	1.12	0.50	0.10	1.50	2.00	2.70	35.60
	Offset_1.5	692	1.97	2.28	1.16	0.02	1.40	1.72	2.27	61.00
	KE_2.0	4,090	2.60	1.71	0.66	0.02	1.90	2.39	3.00	60.30
	Unknown	93,941	0.71	1.17	1.65	0.00	0.25	0.40	0.90	104.00

High grade outliers are values that fall outside the expected statistical distribution of the dataset. It is necessary to identify and treat them to prevent unrealistic high grade spread and ensure that models reflect geologically realistic grade distribution. To limit the influence of high-grade outliers during grade estimation, composites were capped prior to their use in the estimation process. Capping was performed on individual estimation domains (combination of structural, grade, and alteration domains) for metals and density. A combination of histograms and probability plots were used to determine the capping values. Separation of grade populations characterized by inflections in the probability plot or gaps in the high tail of the grade distribution were indicators of potential capping values. Figure 14-18 shows an example of composite data distributions used for capping analysis.

The selected capped values, uncapped and capped composite statistics for gold, copper, and silver are provided in Table 14-11, Table 14-12, and Table 14-13, respectively.

Figure 14-18: Examples of grade capping treatment based on log probability curve



Note: Applied to phyllic and potassic C-S overprint alteration within the low-grade gold domain.

Table 14-11: Summary statistics of length-weighted, capped vs uncapped 2 m gold composites

Statistics weighting: Length-weighted		Uncapped									Capped value	No. samples capped	Capped							
Gold domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	**All**	**113,846**	**0.17**	**0.26**	**1.52**	**0.00**	**0.01**	**0.09**	**0.24**	**12.53**	**-**	**263**	**0.17**	**0.25**	**1.43**	**0.00**	**0.01**	**0.09**	**0.24**	**3.70**
	Nugget_KUG_0.2	**23,572**	**0.35**	**0.21**	**0.62**	**0.00**	**0.23**	**0.31**	**0.43**	**8.35**	**2.50**	**70**	**0.35**	**0.19**	**0.55**	**0.00**	**0.23**	**0.31**	**0.43**	**2.50**
	Overburden	48	0.38	0.20	0.53	0.12	0.26	0.34	0.46	1.32	-	-	0.38	0.20	0.53	0.12	0.26	0.34	0.46	1.32
	Leach-QSP	795	0.30	0.14	0.47	0.00	0.22	0.28	0.36	1.22	0.80	7	0.30	0.13	0.44	0.00	0.22	0.28	0.36	0.80
	Propylitic	920	0.35	0.19	0.54	0.00	0.22	0.32	0.44	2.07	1.00	6	0.34	0.17	0.51	0.00	0.22	0.32	0.44	1.00
	Potassic C-S Overprint	3,492	0.34	0.19	0.57	0.00	0.23	0.31	0.42	3.59	1.25	14	0.34	0.18	0.52	0.00	0.23	0.31	0.42	1.25
	C-S	983	0.40	0.19	0.47	0.00	0.27	0.38	0.50	1.47	1.00	11	0.40	0.18	0.46	0.00	0.27	0.38	0.50	1.00
	Phyllic	6,172	0.33	0.17	0.50	0.00	0.23	0.30	0.40	3.10	1.10	20	0.33	0.15	0.47	0.00	0.23	0.30	0.40	1.10
	Potassic	11,162	0.36	0.25	0.70	0.00	0.23	0.32	0.44	8.35	2.50	12	0.36	0.22	0.61	0.00	0.23	0.32	0.44	2.50
	KUG_0.6	**1,564**	**1.18**	**0.71**	**0.61**	**0.00**	**0.71**	**0.97**	**1.40**	**6.00**	**3.70**	**22**	**1.17**	**0.66**	**0.57**	**0.00**	**0.71**	**0.97**	**1.40**	**3.70**
	Propylitic	108	1.15	0.64	0.56	0.43	0.76	0.98	1.36	5.35	3.00	1	1.12	0.53	0.47	0.43	0.76	0.98	1.36	3.00
	Potassic C-S Overprint	10	0.77	0.16	0.20	0.57	0.69	0.77	0.90	1.08	-	-	0.77	0.16	0.20	0.57	0.69	0.77	0.90	1.08
	C-S	74	0.90	0.36	0.40	0.34	0.62	0.80	1.08	2.08	-	-	0.90	0.36	0.40	0.34	0.62	0.80	1.08	2.08
	Phyllic	749	1.32	0.78	0.59	0.01	0.78	1.08	1.67	6.00	3.50	18	1.31	0.71	0.55	0.01	0.78	1.08	1.67	3.50
	Potassic	623	1.05	0.63	0.60	0.00	0.67	0.88	1.19	4.91	3.70	3	1.04	0.62	0.59	0.00	0.67	0.88	1.19	3.70
	Offset_0.2	**1,018**	**0.29**	**0.17**	**0.60**	**0.00**	**0.19**	**0.26**	**0.36**	**3.29**	**0.80**	**8**	**0.29**	**0.14**	**0.49**	**0.00**	**0.19**	**0.26**	**0.36**	**0.80**
	Phyllic	1,018	0.29	0.17	0.60	0.00	0.19	0.26	0.36	3.29	0.80	8	0.29	0.14	0.49	0.00	0.19	0.26	0.36	0.80
	KE_0.3	**3,075**	**0.46**	**0.19**	**0.41**	**0.00**	**0.34**	**0.44**	**0.55**	**2.41**	**1.50**	**8**	**0.46**	**0.18**	**0.39**	**0.00**	**0.34**	**0.44**	**0.55**	**1.50**
	Propylitic	42	0.37	0.24	0.66	0.00	0.25	0.40	0.55	0.82	-	-	0.37	0.24	0.66	0.00	0.25	0.40	0.55	0.82
	C-S	133	0.39	0.14	0.37	0.14	0.29	0.36	0.47	0.82	-	-	0.39	0.14	0.37	0.14	0.29	0.36	0.47	0.82
	Potassic	2,900	0.47	0.19	0.41	0.00	0.35	0.44	0.55	2.41	1.50	8	0.46	0.18	0.39	0.00	0.35	0.44	0.55	1.50
	KE_0.6	**1,705**	**0.85**	**0.35**	**0.41**	**0.00**	**0.62**	**0.78**	**1.01**	**3.85**	**2.00**	**17**	**0.84**	**0.33**	**0.39**	**0.00**	**0.62**	**0.78**	**1.01**	**2.00**
	Propylitic	18	1.05	0.25	0.24	0.54	0.94	1.07	1.17	1.71	-	-	1.05	0.25	0.24	0.54	0.94	1.07	1.17	1.71
	Potassic	1,687	0.85	0.35	0.41	0.00	0.62	0.77	1.01	3.85	2.00	17	0.84	0.33	0.39	0.00	0.62	0.77	1.01	2.00
	Unknown	**82,912**	**0.08**	**0.14**	**1.77**	**0.00**	**0.01**	**0.04**	**0.11**	**12.53**	**2.20**	**138**	**0.08**	**0.11**	**1.44**	**0.00**	**0.01**	**0.04**	**0.11**	**2.20**
	Overburden	99	0.16	0.13	0.77	0.00	0.07	0.15	0.21	0.50	-	-	0.16	0.13	0.77	0.00	0.07	0.15	0.21	0.50
	Leach-QSP	1,707	0.13	0.07	0.56	0.00	0.08	0.12	0.17	0.67	0.50	6	0.13	0.07	0.55	0.00	0.08	0.12	0.17	0.50
	Propylitic	36,221	0.04	0.12	3.15	0.00	0.00	0.01	0.03	11.54	2.20	11	0.04	0.10	2.52	0.00	0.00	0.01	0.03	2.20
	Advanced Argillic	1,498	0.05	0.07	1.28	0.00	0.01	0.03	0.07	0.71	0.50	2	0.05	0.07	1.25	0.00	0.01	0.03	0.07	0.50
	Potassic C-S Overprint	2,920	0.14	0.13	0.95	0.00	0.06	0.11	0.17	1.81	0.80	17	0.13	0.11	0.84	0.00	0.06	0.11	0.17	0.80
	C-S	8,592	0.07	0.09	1.31	0.00	0.02	0.04	0.09	2.10	0.55	52	0.07	0.08	1.19	0.00	0.02	0.04	0.09	0.55
	Phyllic	14,865	0.12	0.14	1.20	0.00	0.05	0.10	0.15	6.47	1.50	15	0.12	0.11	0.93	0.00	0.05	0.10	0.15	1.50
	Potassic	14,888	0.14	0.18	1.32	0.00	0.05	0.10	0.18	12.53	1.25	32	0.14	0.13	0.98	0.00	0.05	0.10	0.18	1.25
	Unaltered	2,122	0.02	0.08	4.01	0.00	0.00	0.00	0.01	2.26	0.60	3	0.02	0.05	3.02	0.00	0.00	0.00	0.01	0.60

Table 14-12: Summary statistics of length-weighted, capped vs uncapped 2m copper composites

Statistics weighting: Length-weighted		Uncapped									Capped value	No. samples capped	Capped							
Copper domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	**All**	**113,847**	**0.10**	**0.14**	**1.38**	**0.00**	**0.01**	**0.05**	**0.14**	**5.11**	**-**	**1221**	**0.10**	**0.13**	**1.33**	**0.00**	**0.01**	**0.05**	**0.14**	**0.95**
	Nugget_KUG_0.2	**6,981**	**0.27**	**0.14**	**0.52**	**0.00**	**0.20**	**0.25**	**0.32**	**5.11**	**0.70**	**127**	**0.27**	**0.11**	**0.42**	**0.00**	**0.20**	**0.25**	**0.32**	**0.70**
	Propylitic	374	0.28	0.12	0.43	0.00	0.21	0.25	0.32	1.47	0.50	16	0.27	0.09	0.34	0.00	0.21	0.25	0.32	0.50
	Potassic C-S Overprint	638	0.23	0.09	0.41	0.00	0.18	0.22	0.27	0.97	0.40	20	0.22	0.08	0.36	0.00	0.18	0.22	0.27	0.40
	C-S	707	0.27	0.11	0.41	0.00	0.21	0.25	0.31	1.03	0.55	12	0.27	0.10	0.37	0.00	0.21	0.25	0.31	0.55
	Phyllic	1,314	0.28	0.13	0.45	0.00	0.21	0.26	0.33	1.93	0.55	36	0.28	0.10	0.37	0.00	0.21	0.26	0.33	0.55
	Potassic	3,948	0.27	0.16	0.57	0.00	0.19	0.25	0.33	5.11	0.70	43	0.27	0.12	0.45	0.00	0.19	0.25	0.33	0.70
	KUG_0.4	**773**	**0.55**	**0.24**	**0.44**	**0.01**	**0.40**	**0.50**	**0.65**	**1.81**	**0.95**	**57**	**0.54**	**0.20**	**0.37**	**0.01**	**0.40**	**0.50**	**0.65**	**0.95**
	Propylitic	77	0.45	0.13	0.29	0.16	0.36	0.43	0.54	0.79	-	-	0.45	0.13	0.29	0.16	0.36	0.43	0.54	0.79
	C-S	9	0.50	0.08	0.17	0.35	0.48	0.51	0.56	0.60	-	-	0.50	0.08	0.17	0.35	0.48	0.51	0.56	0.60
	Phyllic	529	0.55	0.24	0.45	0.01	0.39	0.49	0.64	1.78	0.95	39	0.53	0.20	0.38	0.01	0.39	0.49	0.64	0.95
	Potassic	158	0.63	0.25	0.40	0.19	0.46	0.58	0.78	1.81	0.95	18	0.61	0.21	0.33	0.19	0.46	0.58	0.78	0.95
	Offset_0.2	**882**	**0.27**	**0.15**	**0.56**	**0.00**	**0.20**	**0.25**	**0.32**	**2.31**	**0.55**	**26**	**0.26**	**0.11**	**0.41**	**0.00**	**0.20**	**0.25**	**0.32**	**0.55**
	Phyllic	822	0.27	0.15	0.56	0.00	0.20	0.25	0.32	2.31	0.55	26	0.26	0.11	0.41	0.00	0.20	0.25	0.32	0.55
	KE_0.2	**4,492**	**0.34**	**0.15**	**0.43**	**0.00**	**0.25**	**0.33**	**0.42**	**1.76**	**0.71**	**106**	**0.34**	**0.13**	**0.39**	**0.00**	**0.25**	**0.33**	**0.42**	**0.71**
	Propylitic	173	0.36	0.17	0.47	0.00	0.27	0.37	0.47	0.84	0.55	20	0.35	0.15	0.43	0.00	0.27	0.37	0.47	0.55
	Advanced Argillic	69	0.27	0.12	0.45	0.00	0.22	0.28	0.34	0.54	-	-	0.27	0.12	0.45	0.00	0.22	0.28	0.34	0.54
	C-S	474	0.33	0.15	0.47	0.02	0.23	0.30	0.40	1.44	0.50	29	0.31	0.11	0.36	0.02	0.23	0.30	0.40	0.50
	Phyllic	87	0.37	0.18	0.48	0.00	0.26	0.42	0.49	0.71	-	-	0.37	0.18	0.48	0.00	0.26	0.42	0.49	0.71
	Potassic	3,665	0.34	0.14	0.42	0.00	0.25	0.33	0.41	1.76	0.70	57	0.34	0.13	0.39	0.00	0.25	0.33	0.41	0.70
	Unaltered	24	0.46	0.10	0.21	0.28	0.39	0.46	0.51	0.63	-	-	0.46	0.10	0.21	0.28	0.39	0.46	0.51	0.63
	KE_0.4	**2,964**	**0.51**	**0.16**	**0.31**	**0.00**	**0.42**	**0.50**	**0.59**	**2.46**	**0.95**	**38**	**0.51**	**0.15**	**0.29**	**0.00**	**0.42**	**0.50**	**0.59**	**0.95**
	Propylitic	29	0.46	0.26	0.57	0.00	0.39	0.56	0.65	0.81	-	-	0.46	0.26	0.57	0.00	0.39	0.56	0.65	0.81
	C-S	30	0.49	0.14	0.28	0.25	0.40	0.44	0.59	0.76	-	-	0.49	0.14	0.28	0.25	0.40	0.44	0.59	0.76
	Potassic	2,905	0.51	0.16	0.31	0.00	0.42	0.50	0.59	2.46	0.95	38	0.51	0.15	0.29	0.00	0.42	0.50	0.59	0.95
	Unknown	**97,815**	**0.06**	**0.08**	**1.27**	**0.00**	**0.01**	**0.03**	**0.09**	**2.51**	**0.35**	**867**	**0.06**	**0.07**	**1.18**	**0.00**	**0.01**	**0.03**	**0.09**	**0.35**
	Overburden	148	0.06	0.06	1.02	0.00	0.02	0.04	0.08	0.30	-	-	0.06	0.06	1.02	0.00	0.02	0.04	0.08	0.30
	Leach-QSP	2,504	0.06	0.06	0.92	0.00	0.02	0.05	0.09	0.61	0.35	8	0.06	0.06	0.90	0.00	0.02	0.05	0.09	0.35
	Propylitic	36,662	0.02	0.06	2.48	0.00	0.00	0.01	0.02	2.51	0.35	260	0.02	0.05	2.18	0.00	0.00	0.01	0.02	0.35
	Advanced Argillic	1,428	0.03	0.06	1.72	0.00	0.00	0.01	0.03	0.46	0.35	7	0.03	0.06	1.70	0.00	0.00	0.01	0.03	0.35
	Potassic C-S Overprint	5,778	0.11	0.07	0.68	0.00	0.05	0.10	0.15	0.71	0.35	52	0.11	0.07	0.66	0.00	0.05	0.10	0.15	0.35
	C-S	8,559	0.05	0.06	1.28	0.00	0.02	0.03	0.05	1.10	0.35	50	0.05	0.05	1.17	0.00	0.02	0.03	0.05	0.35
	Phyllic	20,043	0.09	0.08	0.92	0.00	0.03	0.07	0.12	2.22	0.35	188	0.08	0.07	0.85	0.00	0.03	0.07	0.12	0.35
	Potassic	20,594	0.11	0.08	0.80	0.00	0.04	0.09	0.15	2.18	0.35	301	0.10	0.08	0.73	0.00	0.04	0.09	0.15	0.35
	Unaltered	2,099	0.01	0.03	2.93	0.00	0.00	0.00	0.01	1.12	0.35	1	0.01	0.02	2.05	0.00	0.00	0.00	0.01	0.35

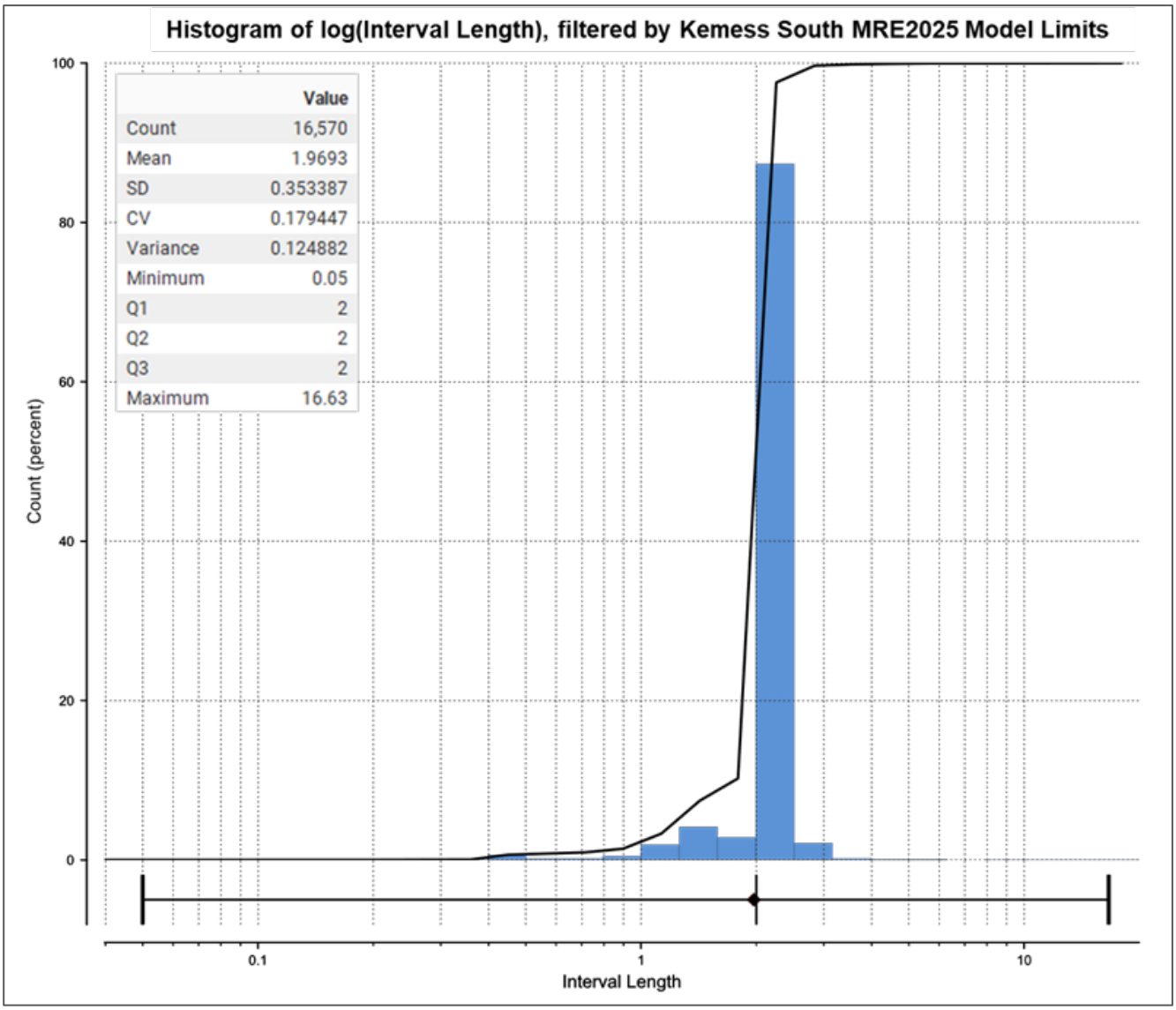
Table 14-13: Summary statistics of length-weighted, capped vs uncapped 2m silver composites

Statistics weighting: Length-weighted			Uncapped								Capped value	No samples capped	Capped							
Silver domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess North	**All**	**96,848**	**0.92**	**1.20**	**1.31**	**0.00**	**0.25**	**0.58**	**1.26**	**68.86**	**-**	**874**	**0.89**	**0.89**	**1.01**	**0.00**	**0.25**	**0.58**	**1.26**	**7.00**
	KUG_1.5	**7,786**	**2.23**	**1.04**	**0.47**	**0.10**	**1.58**	**2.00**	**2.70**	**16.40**	**6.00**	**155**	**2.20**	**0.94**	**0.43**	**0.10**	**1.58**	**2.00**	**2.70**	**6.00**
	C-S	507	2.09	0.84	0.40	0.25	1.59	1.95	2.50	6.30	4.00	19	2.07	0.78	0.37	0.25	1.59	1.95	2.50	4.00
	Leach-QSP	12	2.30	1.08	0.47	0.94	1.46	1.76	2.82	4.28	-	-	2.30	1.08	0.47	0.94	1.46	1.76	2.82	4.28
	Phyllic	2,153	2.45	1.10	0.45	0.25	1.68	2.22	3.10	14.90	5.00	44	2.42	1.00	0.41	0.25	1.68	2.22	3.10	5.00
	Potassic	4,162	2.17	1.05	0.48	0.10	1.54	2.00	2.59	16.40	6.00	39	2.16	0.96	0.44	0.10	1.54	2.00	2.59	6.00
	Potassic C-S Overprint	653	1.83	0.70	0.38	0.10	1.40	1.70	2.10	6.62	3.50	18	1.80	0.61	0.34	0.10	1.40	1.70	2.10	3.50
	Propylitic	299	2.49	1.05	0.42	0.25	1.80	2.38	3.00	10.05	3.50	35	2.38	0.75	0.32	0.25	1.80	2.38	3.00	3.50
	Offset_1.5	**599**	**1.97**	**1.83**	**0.93**	**0.02**	**1.40**	**1.78**	**2.30**	**41.31**	**3.50**	**22**	**1.86**	**0.68**	**0.36**	**0.02**	**1.40**	**1.78**	**2.30**	**3.50**
	Phyllic	599	1.97	1.83	0.93	0.02	1.40	1.78	2.30	41.31	3.50	22	1.86	0.68	0.36	0.02	1.40	1.78	2.30	3.50
	KE_2.0	**3,915**	**2.60**	**1.49**	**0.57**	**0.04**	**2.00**	**2.40**	**3.00**	**41.83**	**7.00**	**67**	**2.54**	**0.96**	**0.38**	**0.04**	**2.00**	**2.40**	**3.00**	**7.00**
	Advanced Argillic	42	2.24	1.20	0.53	0.25	2.00	2.20	2.60	6.50	-	-	2.24	1.20	0.53	0.25	2.00	2.20	2.60	6.50
	C-S	232	2.73	1.28	0.47	0.83	1.90	2.40	3.20	8.15	4.00	30	2.55	0.88	0.34	0.83	1.90	2.40	3.20	4.00
	Phyllic	88	2.69	1.30	0.48	0.29	1.92	2.80	3.52	6.43	-	-	2.69	1.30	0.48	0.29	1.92	2.80	3.52	6.43
	Potassic	3,355	2.57	1.50	0.58	0.25	2.00	2.35	2.90	41.83	7.00	28	2.52	0.94	0.37	0.25	2.00	2.35	2.90	7.00
	Propylitic	174	2.98	1.70	0.57	0.04	2.20	2.93	3.54	16.39	5.00	9	2.85	1.17	0.41	0.04	2.20	2.93	3.54	5.00
	Unaltered	24	2.78	0.59	0.21	1.70	2.44	2.82	3.13	4.33	-	-	2.78	0.59	0.21	1.70	2.44	2.82	3.13	4.33
	Unknown	**84,548**	**0.71**	**1.04**	**1.46**	**0.00**	**0.25**	**0.45**	**0.95**	**68.86**	**4.00**	**630**	**0.68**	**0.67**	**0.98**	**0.00**	**0.25**	**0.45**	**0.95**	**4.00**
	Advanced Argillic	1,455	0.57	0.59	1.03	0.05	0.25	0.27	0.70	6.55	4.00	4	0.57	0.56	0.99	0.05	0.25	0.27	0.70	4.00
	C-S	8,804	0.93	1.47	1.58	0.03	0.27	0.62	1.10	52.50	4.00	169	0.86	0.78	0.91	0.03	0.27	0.62	1.10	4.00
	Leach-QSP	1,561	0.49	0.49	0.99	0.10	0.25	0.37	0.63	8.27	4.00	5	0.48	0.42	0.86	0.10	0.25	0.37	0.63	4.00
	Phyllic	14,164	0.74	0.75	1.01	0.01	0.30	0.62	1.00	41.25	4.00	48	0.73	0.55	0.76	0.01	0.30	0.62	1.00	4.00
	Potassic	19,270	1.01	0.92	0.91	0.02	0.46	0.84	1.39	61.50	4.00	136	1.00	0.72	0.72	0.02	0.46	0.84	1.39	4.00
	Potassic C-S Overprint	4,374	0.90	0.68	0.76	0.04	0.45	0.77	1.16	9.77	4.00	23	0.89	0.64	0.71	0.04	0.45	0.77	1.16	4.00
	Propylitic	33,062	0.48	1.10	2.28	0.00	0.18	0.25	0.44	68.86	4.00	243	0.45	0.59	1.31	0.00	0.18	0.25	0.44	4.00
	Unaltered	1,748	0.24	0.65	2.66	0.05	0.10	0.15	0.25	21.56	4.00	2	0.23	0.30	1.31	0.05	0.10	0.15	0.25	4.00
	Overburden	110	0.62	0.52	0.84	0.02	0.25	0.50	0.87	2.62	-	-	0.62	0.52	0.84	0.02	0.25	0.50	0.87	2.62

14.6.2 Kemess South

For the Kemess South deposit, a similar approach to Kemess Main Zone was taken for capping and compositing. Over 90% of assays were sampled at 2 m intervals or less (Figure 14-19). Assay data were composited to 2 m composites, with a minimum sample coverage of 1 m. Residual end lengths shorter than 1 m were distributed equally throughout the drillhole. Compositing was performed within the boundaries of mineralization respecting fault boundaries.

Figure 14-19: Sample length distribution and associated statistics



A summary of length-weighted gold, copper, and silver assays within the mineralized domains can be found in Table 14-14, Table 14-15, and Table 14-16 respectively.

Table 14-14: Summary statistics of length-weighted gold assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	All	17,004	0.26	0.35	1.34	0.00	0.00	0.13	0.39	7.12
	KS_0.2_Au	9,446	0.55	0.36	0.65	0.00	0.30	0.45	0.72	7.12
	Unknown	7,091	0.09	0.10	1.09	0.00	0.02	0.08	0.14	2.86

Table 14-15: Summary statistics of length-weighted copper assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	All	17,004	0.10	0.17	1.74	0.00	0.00	0.06	0.15	10.40
	KS_0.2_Cu	3,161	0.31	0.27	0.89	0.01	0.21	0.26	0.33	10.40
	Unknown	13,376	0.10	0.14	1.42	0.00	0.03	0.08	0.14	5.71

Table 14-16: Summary statistics of length-weighted silver assays

	Au domain	Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	All	12,995	0.45	0.65	1.44	0.00	0.00	0.10	0.70	14.30
	KS_1.0_Ag	10,048	0.49	0.56	1.14	0.10	0.10	0.40	0.70	14.30
	Unknown	2,480	1.42	0.67	0.47	0.10	1.10	1.30	1.60	12.40

To limit the influence of high-grade outliers during grade estimation, composites were capped prior to their use in the estimation process. Capping was performed on individual estimation domains for metals (combination of structural and grade domains) and density (within lithology domains). A combination of histograms and probability plots were used to determine the capping values. Figure 14-20 shows an example of composite data distributions used for capping analysis.

The selected capped values, uncapped and capped composite statistics for gold, copper, and silver are provided in Table 14-17, Table 14-18, and Table 14-19 respectively.

The lower assay counts shown in Table 14-14 to Table 14-16 when compared to composite counts are due to the treatment of unsampled assay intervals. At Kemess South, a large portion of the Hazelton formation was unsampled as it is known to be barren. For compositing, all unsampled intervals were assumed to be barren and assigned a nominal value of ¼ of the detection limit. Including these intervals increases the total the number of composites.

Figure 14-20: Composite data distribution used for the determination of capping values of the main gold domain of Kemess South (within Fault Block 3)

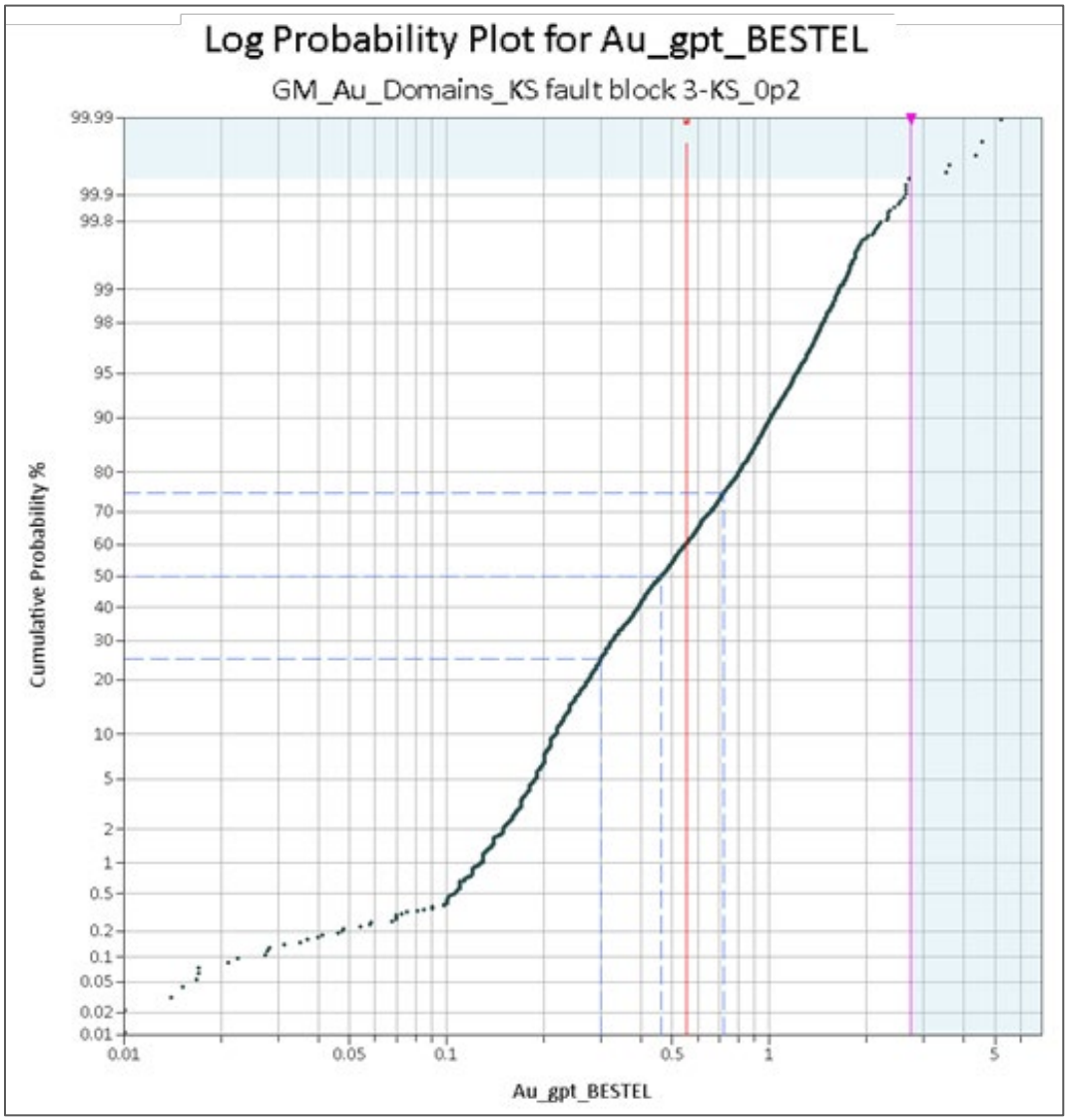


Table 14-17: Summary statistics of length-weighted, capped vs uncapped 2 m gold composites

Statistics weighting: Length-weighted			Uncapped								Capped value	No. samples capped	Capped							
Gold domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	**All**	**24,003**	**0.24**	**0.34**	**1.38**	**0.00**	**0.00**	**0.11**	**0.37**	**6.99**	**2.75**	**72**	**0.24**	**0.33**	**1.36**	**0.00**	**0.00**	**0.11**	**0.37**	**2.75**
	KS_0.2_Au_FB1	**130**	**0.27**	**0.10**	**0.37**	**0.07**	**0.20**	**0.25**	**0.33**	**0.58**	**2.75**	**-**	**0.27**	**0.10**	**0.37**	**0.07**	**0.20**	**0.25**	**0.33**	**0.58**
	Black Lake	130	0.27	0.10	0.37	0.07	0.20	0.25	0.33	0.58	2.75	-	0.27	0.10	0.37	0.07	0.20	0.25	0.33	0.58
	KS_UNK_Au_FB1	**733**	**0.04**	**0.07**	**1.86**	**0.00**	**0.00**	**0.01**	**0.02**	**0.49**	**0.30**	**5**	**0.04**	**0.07**	**1.82**	**0.00**	**0.00**	**0.01**	**0.02**	**0.30**
	Asitka	338	0.00	0.01	1.71	0.00	0.00	0.00	0.01	0.08	-	-	0.00	0.01	1.71	0.00	0.00	0.00	0.01	0.08
	Takla	47	0.08	0.06	0.76	0.01	0.02	0.08	0.11	0.25	-	-	0.08	0.06	0.76	0.01	0.02	0.08	0.11	0.25
	Black Lake	131	0.16	0.08	0.49	0.00	0.12	0.15	0.21	0.49	0.30	4	0.16	0.07	0.43	0.00	0.12	0.15	0.21	0.30
	Hazelton	93	0.01	0.03	2.60	0.00	0.01	0.01	0.01	0.17	-	-	0.01	0.03	2.60	0.00	0.01	0.01	0.01	0.17
	Overburden	124	0.01	0.04	4.17	0.00	0.00	0.00	0.00	0.33	0.30	1	0.01	0.04	4.06	0.00	0.00	0.00	0.00	0.30
	KS_UNK_Au_FB2	**1,311**	**0.03**	**0.10**	**3.26**	**0.00**	**0.00**	**0.01**	**0.02**	**1.58**	**0.60**	**8**	**0.03**	**0.07**	**2.69**	**0.00**	**0.00**	**0.01**	**0.02**	**0.60**
	Asitka	1,030	0.03	0.10	3.48	0.00	0.00	0.01	0.02	1.58	0.60	6	0.03	0.07	2.79	0.00	0.00	0.01	0.02	0.60
	Takla	3	0.02	0.01	0.33	0.01	0.01	0.02	0.02	0.02	-	-	0.02	0.01	0.33	0.01	0.01	0.02	0.02	0.02
	Black Lake	1	0.01	-	-	0.01	0.01	0.01	0.01	0.01	-	-	0.01	-	-	0.01	0.01	0.01	0.01	0.01
	Epiclastic	170	0.05	0.10	2.14	0.00	0.00	0.00	0.05	0.75	0.60	2	0.05	0.09	2.02	0.00	0.00	0.00	0.05	0.60
	Overburden	107	0.01	0.02	2.07	0.00	0.00	0.00	0.01	0.15	-	-	0.01	0.02	2.07	0.00	0.00	0.00	0.01	0.15
	KS_0.2_Au_FB3	**9,347**	**0.55**	**0.35**	**0.64**	**0.00**	**0.30**	**0.46**	**0.72**	**6.99**	**2.75**	**6**	**0.56**	**0.34**	**0.62**	**0.00**	**0.30**	**0.47**	**0.73**	**2.75**
	Takla	142	0.32	0.16	0.50	0.08	0.21	0.27	0.38	0.90	-	-	0.32	0.16	0.50	0.08	0.21	0.27	0.38	0.90
	Black Lake	9,115	0.56	0.35	0.63	0.00	0.30	0.47	0.73	6.99	2.75	5	0.56	0.34	0.62	0.00	0.30	0.47	0.73	2.75
	Hazelton	77	0.57	0.57	1.00	0.18	0.26	0.43	0.68	4.57	2.75	1	0.55	0.42	0.77	0.18	0.26	0.43	0.68	2.75
	Epiclastic	11	0.30	0.12	0.41	0.13	0.20	0.27	0.41	0.51	-	-	0.30	0.12	0.41	0.13	0.20	0.27	0.41	0.51
	Overburden	2	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	-	-	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	KS_UNK_Au_FB3	**12,482**	**0.05**	**0.08**	**1.66**	**0.00**	**0.00**	**0.00**	**0.09**	**2.86**	**0.60**	**17**	**0.05**	**0.07**	**1.52**	**0.00**	**0.00**	**0.00**	**0.09**	**0.60**
	Asitka	93	0.01	0.01	1.19	0.00	0.00	0.01	0.01	0.10	-	-	0.01	0.01	1.19	0.00	0.00	0.01	0.01	0.10
	Takla	3,092	0.08	0.07	0.84	0.00	0.03	0.07	0.11	0.99	0.60	2	0.08	0.06	0.81	0.00	0.03	0.07	0.11	0.60
	Black Lake	2,369	0.14	0.11	0.75	0.00	0.09	0.13	0.18	2.86	0.60	14	0.14	0.08	0.58	0.00	0.09	0.13	0.18	0.60
	Hazelton	5,657	0.00	0.01	3.96	0.00	0.00	0.00	0.00	0.32	-	-	0.00	0.01	3.96	0.00	0.00	0.00	0.00	0.32
	Epiclastic	132	0.03	0.07	2.25	0.00	0.00	0.00	0.02	0.64	0.60	1	0.03	0.07	2.21	0.00	0.00	0.00	0.02	0.60
	Overburden	1,139	0.00	0.01	7.23	0.00	0.00	0.00	0.00	0.19	-	-	0.00	0.01	7.23	0.00	0.00	0.00	0.00	0.19

Table 14-18: Summary statistics of length-weighted, capped vs uncapped 2 m copper composites

Statistics weighting: Length-weighted			Uncapped								Capped value	No. samples capped	Capped							
Copper domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	**All**	**23,627**	**0.10**	**0.16**	**1.72**	**0.00**	**0.00**	**0.05**	**0.14**	**10.40**	**1.60**	**106**	**0.09**	**0.13**	**1.43**	**0.00**	**0.00**	**0.05**	**0.14**	**1.60**
	KS_0.2_Cu_FB1	**847**	**0.02**	**0.03**	**1.48**	**0.00**	**0.00**	**0.00**	**0.03**	**0.26**	**-**	**-**	**0.02**	**0.03**	**1.48**	**0.00**	**0.00**	**0.00**	**0.03**	**0.26**
	Asitka	338	0.00	0.01	1.78	0.00	0.00	0.00	0.01	0.08	-	-	0.00	0.01	1.78	0.00	0.00	0.00	0.01	0.08
	Takla	47	0.03	0.02	0.70	0.00	0.01	0.03	0.05	0.08	-	-	0.03	0.02	0.70	0.00	0.01	0.03	0.05	0.08
	Black Lake	245	0.05	0.03	0.62	0.00	0.03	0.04	0.07	0.26	-	-	0.05	0.03	0.62	0.00	0.03	0.04	0.07	0.26
	Hazelton	93	0.00	0.01	3.70	0.00	0.00	0.00	0.00	0.10	-	-	0.00	0.01	3.70	0.00	0.00	0.00	0.00	0.10
	Overburden	124	0.00	0.01	1.41	0.00	0.00	0.00	0.01	0.03	-	-	0.00	0.01	1.41	0.00	0.00	0.00	0.01	0.03
	KS_UNK_Cu_FB2	**1,311**	**0.06**	**0.28**	**4.53**	**0.00**	**0.00**	**0.00**	**0.01**	**3.11**	**1.00**	**24**	**0.05**	**0.16**	**3.56**	**0.00**	**0.00**	**0.00**	**0.01**	**1.00**
	Asitka	1,030	0.03	0.16	5.62	0.00	0.00	0.00	0.01	2.61	1.00	3	0.02	0.09	3.98	0.00	0.00	0.00	0.01	1.00
	Takla	3	0.01	0.00	0.26	0.01	0.01	0.01	0.01	0.01	-	-	0.01	0.00	0.26	0.01	0.01	0.01	0.01	0.01
	Black Lake	1	0.00	-	-	0.00	0.00	0.00	0.00	0.00	-	-	0.00	-	-	0.00	0.00	0.00	0.00	0.00
	Epiclastic	170	0.25	0.59	2.39	0.00	0.00	0.00	0.12	3.11	1.00	19	0.17	0.32	1.95	0.00	0.00	0.00	0.12	1.00
	Overburden	107	0.08	0.27	3.44	0.00	0.00	0.00	0.01	1.97	1.00	2	0.07	0.21	3.08	0.00	0.00	0.00	0.01	1.00
	KS_0.2_Cu_FB3	**3,222**	**0.30**	**0.27**	**0.89**	**0.00**	**0.21**	**0.26**	**0.33**	**10.40**	**1.60**	**15**	**0.29**	**0.18**	**0.62**	**0.00**	**0.21**	**0.26**	**0.33**	**1.60**
	Takla	43	0.30	0.15	0.50	0.08	0.21	0.27	0.37	0.96	-	-	0.30	0.15	0.50	0.08	0.21	0.27	0.37	0.96
	Black Lake	3,112	0.30	0.27	0.88	0.00	0.21	0.26	0.33	10.40	1.60	15	0.30	0.18	0.60	0.00	0.21	0.26	0.33	1.60
	Hazelton	9	0.26	0.12	0.46	0.07	0.17	0.27	0.30	0.44	-	-	0.26	0.12	0.46	0.07	0.17	0.27	0.30	0.44
	Epiclastic	5	0.30	0.21	0.69	0.03	0.18	0.27	0.48	0.53	-	-	0.30	0.21	0.69	0.03	0.18	0.27	0.48	0.53
	Overburden	53	0.04	0.21	5.88	0.00	0.00	0.00	0.00	1.50	-	-	0.04	0.21	5.88	0.00	0.00	0.00	0.00	1.50
	KS_UNK_Cu_FB3	**18,247**	**0.07**	**0.09**	**1.39**	**0.00**	**0.00**	**0.04**	**0.11**	**3.08**	**1.00**	**14**	**0.07**	**0.08**	**1.29**	**0.00**	**0.00**	**0.04**	**0.11**	**1.00**
	Asitka	93	0.00	0.00	0.51	0.00	0.00	0.00	0.01	0.01	-	-	0.00	0.00	0.51	0.00	0.00	0.00	0.01	0.01
	Takla	3,172	0.06	0.05	0.96	0.00	0.02	0.04	0.08	0.86	-	-	0.06	0.05	0.96	0.00	0.02	0.04	0.08	0.86
	Black Lake	8,300	0.12	0.10	0.81	0.00	0.06	0.11	0.16	3.08	1.00	11	0.12	0.09	0.73	0.00	0.06	0.11	0.16	1.00
	Hazelton	5,499	0.00	0.03	7.75	0.00	0.00	0.00	0.00	0.98	-	-	0.00	0.03	7.75	0.00	0.00	0.00	0.00	0.98
	Epiclastic	141	0.09	0.26	2.73	0.00	0.00	0.00	0.05	1.67	1.00	2	0.08	0.20	2.44	0.00	0.00	0.00	0.05	1.00
	Overburden	1,042	0.00	0.07	18.91	0.00	0.00	0.00	0.00	2.17	1.00	1	0.00	0.03	13.05	0.00	0.00	0.00	0.00	1.00

Table 14-19: Summary statistics of length-weighted, capped vs uncapped 2 m silver composites

Statistics weighting: Length-weighted			Uncapped								Capped value	No. samples capped	Capped							
Silver domains		Count	Mean	Std Dev	CV	Min	25th	50th	75th	Max			Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess South	**All**	**19,623**	**0.46**	**0.78**	**1.70**	**0.00**	**0.00**	**0.17**	**0.70**	**62.85**	**4.00**	**62**	**0.45**	**0.59**	**1.31**	**0.00**	**0.00**	**0.17**	**0.70**	**4.00**
	KS_UKN_Ag_FB1	**738**	**0.11**	**0.13**	**1.22**	**0.00**	**0.02**	**0.10**	**0.10**	**1.40**	**-**	**-**	**0.11**	**0.13**	**1.22**	**0.00**	**0.02**	**0.10**	**0.10**	**1.40**
	Asitka	268	0.10	0.16	1.53	0.00	0.00	0.08	0.10	0.80	-	-	0.10	0.16	1.53	0.00	0.00	0.08	0.10	0.80
	Takla	47	0.12	0.05	0.42	0.10	0.10	0.10	0.10	0.30	-	-	0.12	0.05	0.42	0.10	0.10	0.10	0.10	0.30
	Black Lake	245	0.14	0.14	1.01	0.10	0.10	0.10	0.10	1.40	-	-	0.14	0.14	1.01	0.10	0.10	0.10	0.10	1.40
	Hazelton	93	0.10	0.08	0.81	0.00	0.10	0.10	0.10	0.45	-	-	0.10	0.08	0.81	0.00	0.10	0.10	0.10	0.45
	Overburden	85	0.04	0.05	1.23	0.00	0.00	0.00	0.10	0.24	-	-	0.04	0.05	1.23	0.00	0.00	0.00	0.10	0.24
	KS_UNK_Ag_FB2	**1,170**	**0.33**	**0.54**	**1.63**	**0.00**	**0.00**	**0.10**	**0.45**	**5.02**	**2.50**	**4**	**0.33**	**0.52**	**1.59**	**0.00**	**0.00**	**0.10**	**0.45**	**2.50**
	Asitka	889	0.32	0.49	1.53	0.00	0.00	0.10	0.51	2.96	2.50	3	0.32	0.49	1.53	0.00	0.00	0.10	0.51	2.50
	Takla	3	1.88	1.14	0.61	0.92	0.92	1.60	3.06	3.06	2.50	1	1.68	0.83	0.50	0.92	0.92	1.60	2.50	2.50
	Black Lake	1	0.46	-	-	0.46	0.46	0.46	0.46	0.46	-	-	0.46	-	-	0.46	0.46	0.46	0.46	0.46
	Epiclastic	170	0.19	0.47	2.43	0.00	0.00	0.08	0.20	5.02	2.50	1	0.18	0.34	1.89	0.00	0.00	0.08	0.20	2.50
	Overburden	107	0.61	0.81	1.32	0.00	0.00	0.10	1.40	2.40	-	-	0.61	0.81	1.32	0.00	0.00	0.10	1.40	2.40
	KS_1.0_Ag_FB3	**2,460**	**1.41**	**0.66**	**0.46**	**0.00**	**1.10**	**1.30**	**1.60**	**12.40**	**4.00**	**14**	**1.40**	**0.52**	**0.37**	**0.00**	**1.10**	**1.30**	**1.60**	**4.00**
	Takla	173	1.54	0.49	0.32	0.10	1.23	1.48	1.88	2.60	-	-	1.54	0.49	0.32	0.10	1.23	1.48	1.88	2.60
	Black Lake	2,151	1.41	0.68	0.48	0.00	1.10	1.30	1.60	12.40	4.00	14	1.40	0.52	0.38	0.00	1.10	1.30	1.60	4.00
	Hazelton	136	1.27	0.40	0.31	0.62	1.10	1.20	1.35	3.60	-	-	1.27	0.40	0.31	0.62	1.10	1.20	1.35	3.60
	KS_UNK_Ag_FB3	**15,255**	**0.33**	**0.71**	**2.17**	**0.00**	**0.00**	**0.10**	**0.50**	**62.85**	**4.00**	**13**	**0.32**	**0.46**	**1.43**	**0.00**	**0.00**	**0.10**	**0.50**	**4.00**
	Asitka	93	0.49	0.62	1.28	0.05	0.05	0.20	0.55	2.65	-	-	0.49	0.62	1.28	0.05	0.05	0.20	0.55	2.65
	Takla	2,392	0.58	1.40	2.40	0.00	0.20	0.40	0.70	62.85	4.00	1	0.55	0.54	0.98	0.00	0.20	0.40	0.70	4.00
	Black Lake	6,184	0.50	0.54	1.06	0.00	0.10	0.40	0.70	10.87	4.00	9	0.50	0.48	0.95	0.00	0.10	0.40	0.70	4.00
	Hazelton	5,347	0.08	0.30	3.63	0.00	0.00	0.00	0.00	11.26	4.00	2	0.08	0.26	3.16	0.00	0.00	0.00	0.00	4.00
	Epiclastic	145	0.15	0.32	2.18	0.00	0.00	0.00	0.10	2.00	-	-	0.15	0.32	2.18	0.00	0.00	0.00	0.10	2.00
	Overburden	1,094	0.01	0.06	6.50	0.00	0.00	0.00	0.00	1.19	-	-	0.01	0.06	6.50	0.00	0.00	0.00	0.00	1.19

14.7 SPECIFIC GRAVITY

14.7.1 Kemess Main Zone

The Kemess database contains 52,722 SG measurements. Within the Kemess Main Zone block model boundary and excluding outliers where sample lengths were greater than 10 m, 51,065 measurements were used in the resource estimate. All values were determined by company geologist from whole drill core in the field using the water immersion method. Historically, sample data were collected from all assay intervals. Since 2019, SG measurements have been taken from approximately one 10 cm drill core sample for every 10 m interval along the entire length of each drill hole (Table 14-20). Figure 14-21 shows the spatial distribution of SG measurements used in the current resource estimation. A summary of length-weighted SG measurements can be found in Table 14-21.

Table 14-20: SG measurement methodology in the Kemess District Project

Campaign	Count	Method	Interval	Material
Historical	47,825	Standard water immersion method (no wax coating)	15-20 cm long; every assay interval	Whole core samples

Figure 14-21: Long section view showing the distribution of SG measurements in the Kemess Main Zone deposit

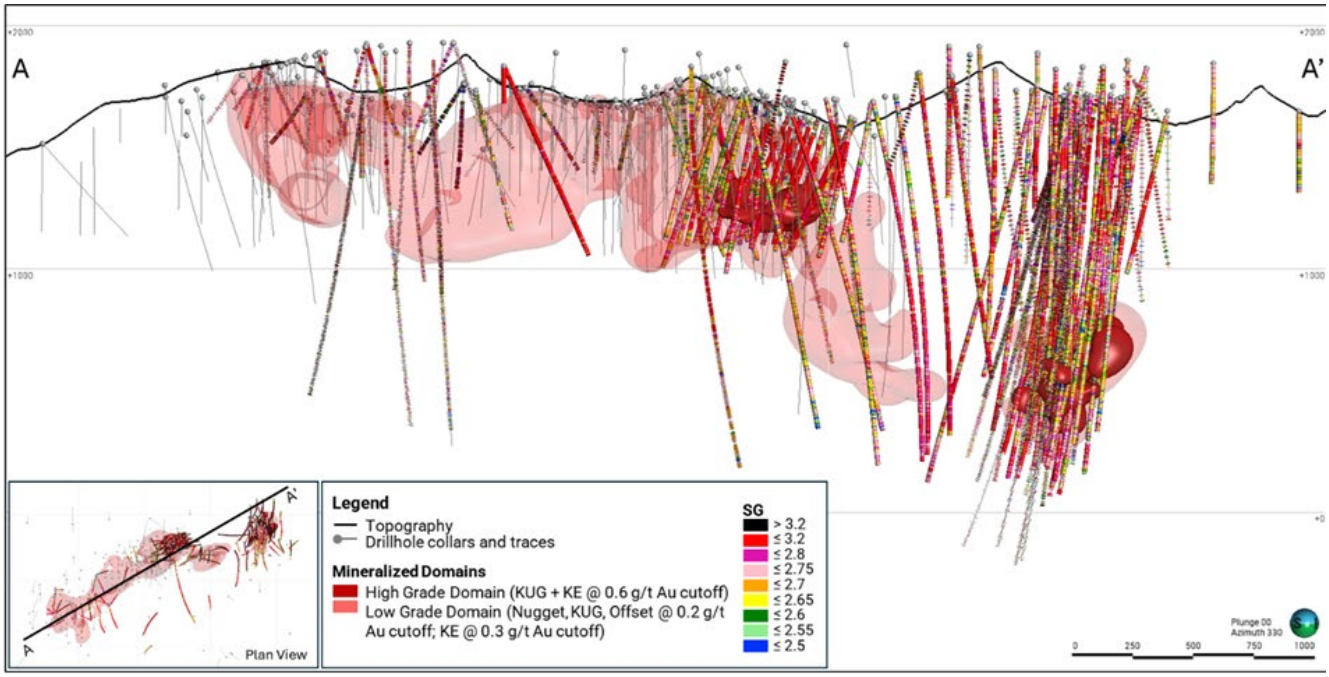


Table 14-21: Statistics of SG measurements used in the Kemess Main Zone MRE

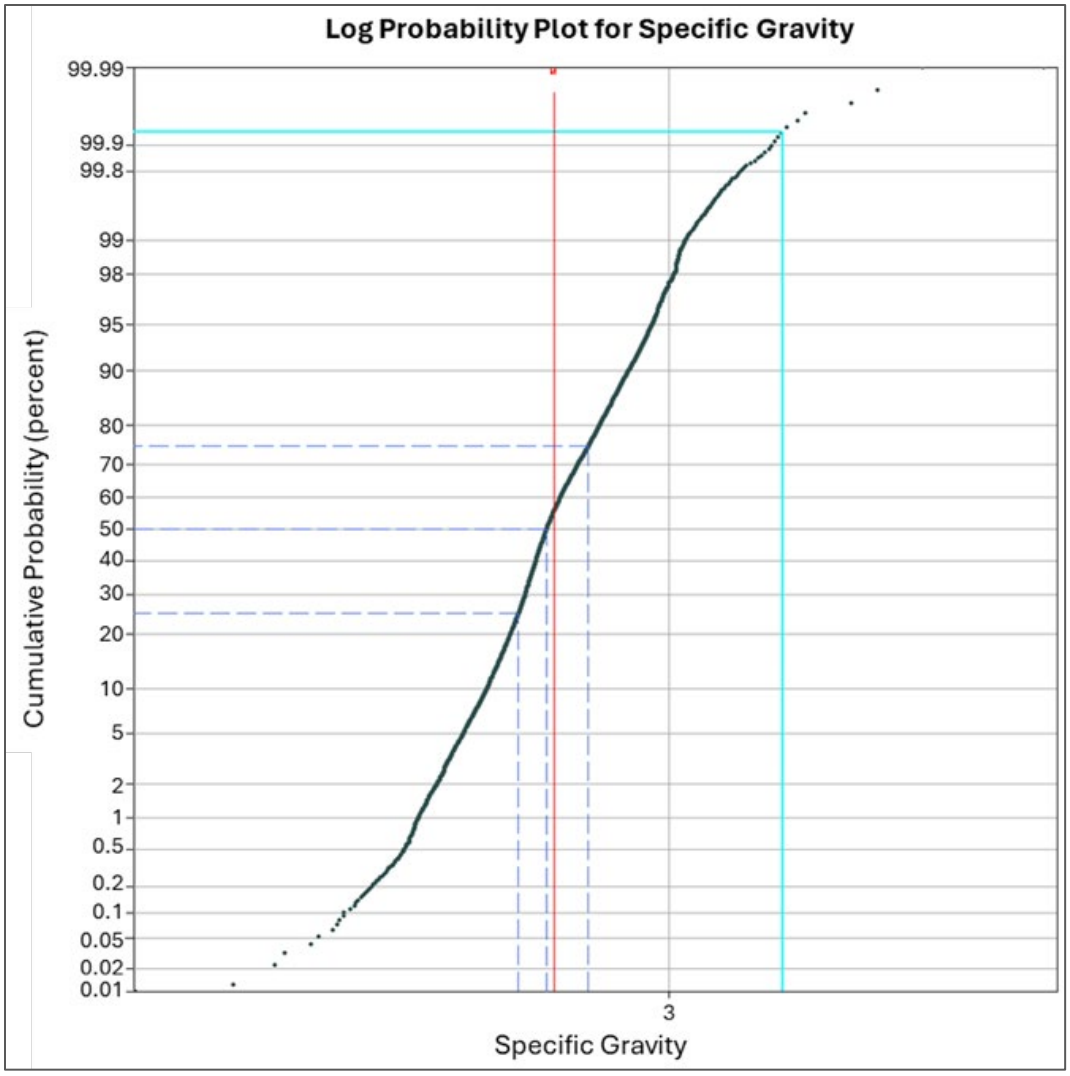
SG Assays									
Domains	**Count**	**Mean**	**Std Dev**	**CV**	**Min**	**25th**	**50th**	**75th**	**Max**
SG All	**51,065**	**2.75**	**0.13**	**0.05**	**2.01**	**2.67**	**2.74**	**2.82**	**4.07**
Advanced Argillic	1,195	2.78	0.10	0.04	2.38	2.71	2.79	2.84	3.26
C-S	5,524	2.79	0.12	0.04	2.22	2.71	2.79	2.87	3.35
Leach-QSP	664	2.67	0.15	0.06	2.04	2.58	2.66	2.76	3.33
Overburden	17	2.75	0.12	0.04	2.30	2.61	2.79	2.81	2.94
Phyllic	7,761	2.78	0.13	0.05	2.07	2.70	2.79	2.87	3.74
Potassic	10,764	2.70	0.11	0.04	2.10	2.63	2.71	2.77	3.80
Potassic C-S Overprint	1,197	2.75	0.14	0.05	2.35	2.64	2.77	2.85	3.25
Propylitic	23,350	2.75	0.13	0.05	2.01	2.68	2.74	2.81	4.07
Unaltered	591	2.67	0.09	0.03	2.12	2.64	2.69	2.72	3.44
*Inactive	2	2.69	0.04	0.02	2.66	2.66	2.66	2.72	2.72

SG measurements were composited and capped to ensure good sample support and limit unbiased tonnage estimation. SG measurements were composited to 2 m intervals and using histograms and probability plots of the entire 2 m composite dataset it was determined reasonable to cap the data at 3.2 to constrain high values (Figure 14-22). Interpolation of SG was constrained within alteration domains using the Nearest Neighbour (NN) search. Alteration was selected as the primary control as statistical evaluation indicated that density variations correlated more strongly with alteration intensity than lithological boundaries in this deposit. Uncapped and capped composite statistics for density is provided in Table 14-22.

Table 14-22: Statistics of 2 m SG composites – capped vs uncapped

Domains		Count	Uncapped								Capped							
			Mean	Std Dev	CV	Min	25th	50th	75th	Max	Mean	Std Dev	CV	Min	25th	50th	75th	Max
Kemess Main Zone	**SG All**	**50,757**	**2.74**	**0.12**	**0.04**	**2.04**	**2.67**	**2.74**	**2.82**	**3.77**	**2.74**	**0.12**	**0.04**	**2.04**	**2.67**	**2.74**	**2.82**	**3.20**
	Advanced Argillic	1,214	2.78	0.10	0.03	2.38	2.72	2.79	2.84	3.15	2.78	0.10	0.03	2.38	2.72	2.79	2.84	3.15
	C-S	5,670	2.78	0.12	0.04	2.25	2.71	2.78	2.86	3.30	2.78	0.12	0.04	2.25	2.71	2.78	2.86	3.20
	Leach-QSP	615	2.66	0.14	0.05	2.04	2.58	2.66	2.74	3.23	2.66	0.14	0.05	2.04	2.58	2.66	2.74	3.20
	Overburden	25	2.70	0.15	0.06	2.30	2.61	2.72	2.81	2.97	2.70	0.15	0.06	2.30	2.61	2.72	2.81	2.97
	Phyllic	7,434	2.78	0.13	0.05	2.14	2.69	2.79	2.87	3.57	2.78	0.13	0.04	2.14	2.69	2.79	2.87	3.20
	Potassic	11,196	2.70	0.11	0.04	2.10	2.63	2.71	2.77	3.70	2.70	0.11	0.04	2.10	2.63	2.71	2.77	3.20
	Potassic C-S Overprint	1,172	2.75	0.13	0.05	2.35	2.65	2.76	2.85	3.21	2.75	0.13	0.05	2.35	2.65	2.76	2.85	3.20
	Propylitic	22,831	2.74	0.12	0.04	2.13	2.68	2.74	2.81	3.77	2.74	0.12	0.04	2.13	2.68	2.74	2.81	3.20
	Unaltered	579	2.67	0.08	0.03	2.34	2.64	2.68	2.71	3.44	2.67	0.08	0.03	2.34	2.64	2.68	2.71	3.20
	*Inactive	21	2.64	0.11	0.04	2.37	2.61	2.66	2.71	2.85	2.64	0.11	0.04	2.37	2.61	2.66	2.71	2.85

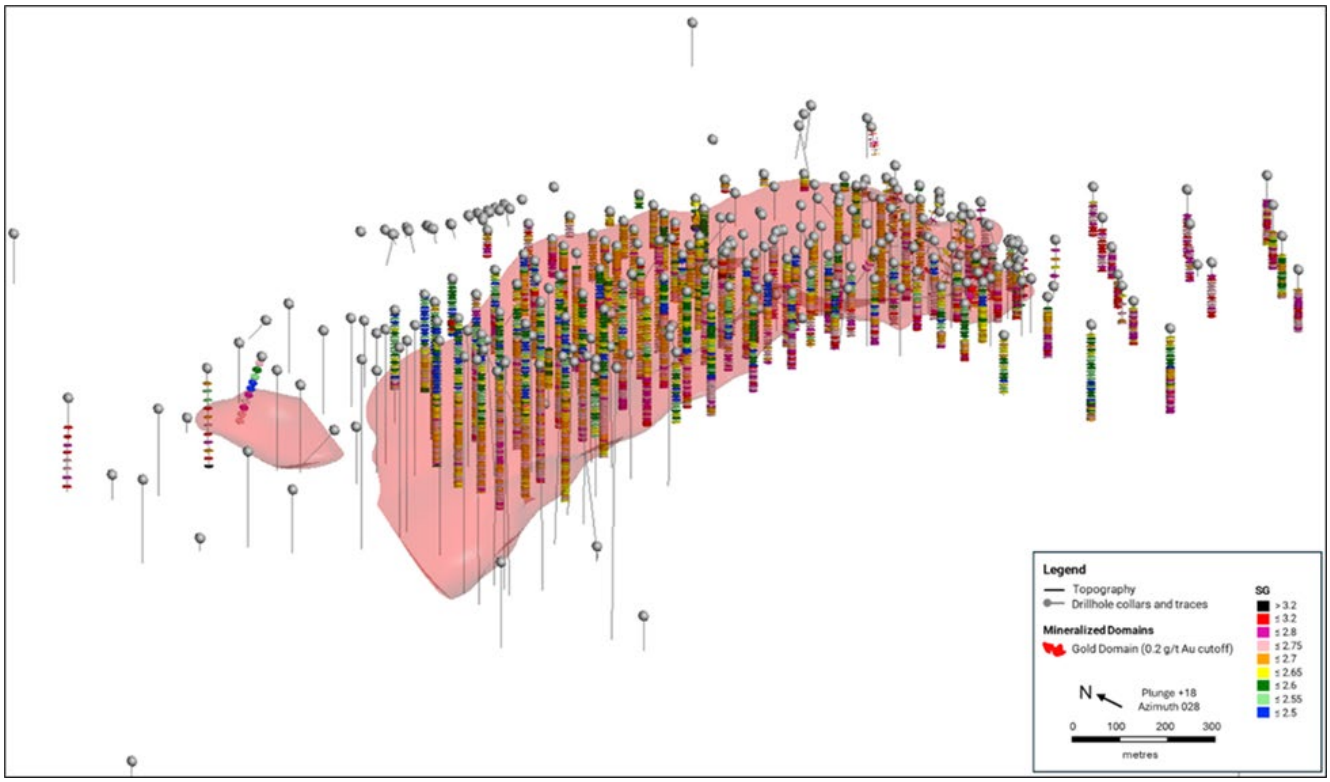
Figure 14-22: Log probability plot of 2 m SG composites (a value of 3.2 was used to cap composites)



14.7.2 Kemess South

The SG database at Kemess South includes 3,511 SG measurements. 3,161 SG measurements from 1991, 1994, 2006, and 2007 fall within the Kemess South block model boundary and were used for the resource estimate. Most of the SG measurements were done on pulps from the El Condor 1991 drilling program. In 2025, field SG measurements were completed on whole drill core using the water immersion method. Results were comparable to the 1991 data. Figure 14-23 shows the spatial distribution of SG measurements used in the current resource estimation.

Figure 14-23: Oblique view showing the distribution of SG measurements in the Kemess South deposit



SG measurements were composited and capped to ensure good sample support and limit unbiased tonnage estimation. Using histograms and probability plots of the entire 2 m composite dataset, SG measurements were composited to 2 m intervals and capped to constrain high values. Statistical analysis indicated the cutoff at an SG of 3 (Figure 14-24). Interpolation of SG was constrained within lithological domains using the NN search. A summary of length-weighted SG measurements can be found in Table 14-23. Uncapped and capped composite statistics for density is provided in Table 14-24.

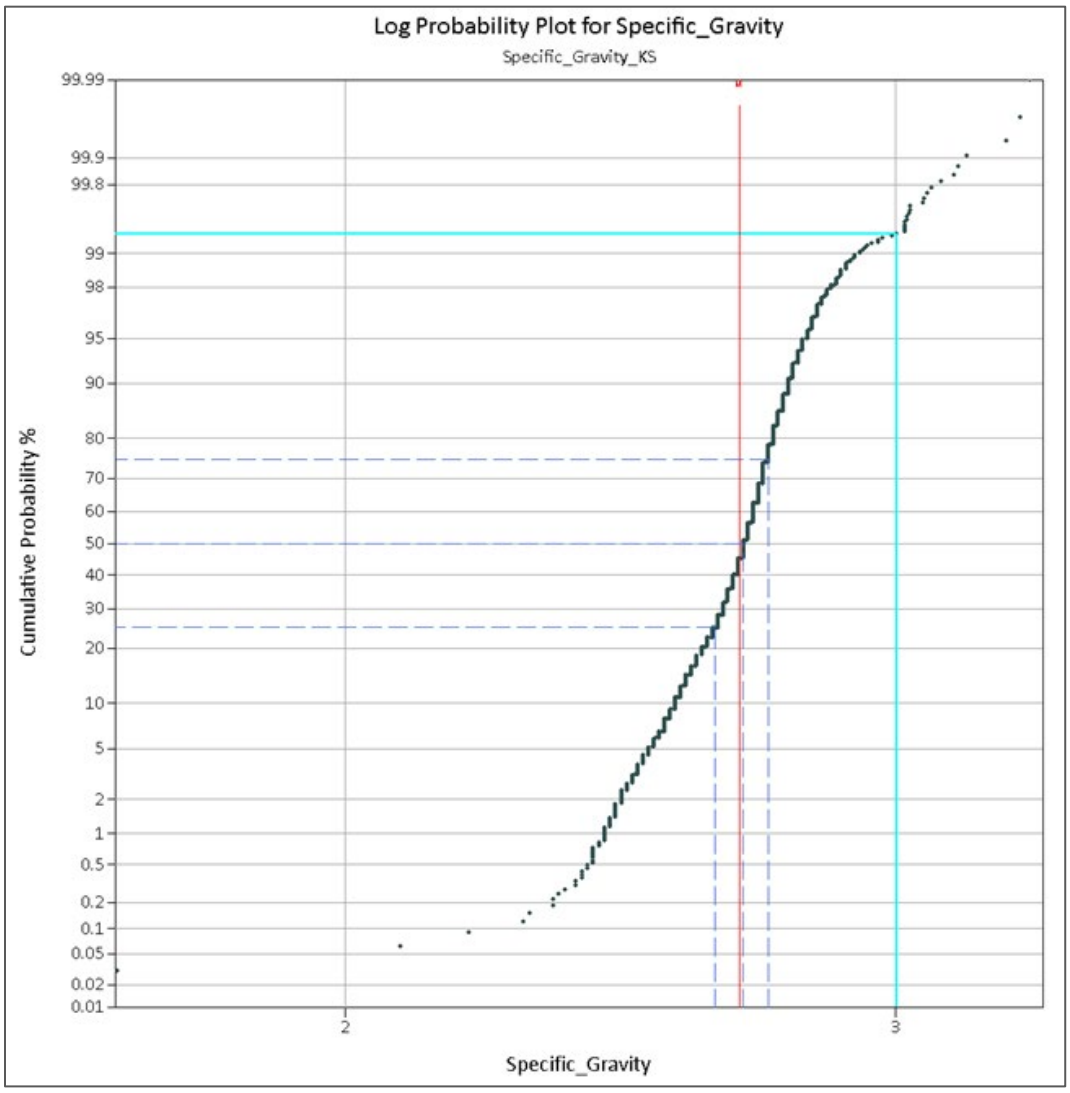
Table 14-23: Statistics of SG measurements used in the Kemess South MRE

SG Assays									
Domains	**Count**	**Mean**	**Std Dev**	**CV**	**Min**	**25th**	**50th**	**75th**	**Max**
SG All	**3,161**	**2.67**	**0.10**	**0.04**	**1.69**	**2.62**	**2.68**	**2.73**	**3.49**
Asitka	98	2.70	0.11	0.04	2.48	2.63	2.71	2.75	3.16
Takla	271	2.76	0.08	0.03	2.47	2.71	2.75	2.80	3.29
Black Lake	2,132	2.70	0.07	0.03	2.08	2.66	2.69	2.73	3.49
Hazelton	634	2.55	0.09	0.03	1.69	2.51	2.56	2.60	2.92
Epiclastic	20	2.65	0.11	0.04	2.42	2.60	2.64	2.72	2.91
Overburden	6	2.72	0.03	0.01	2.68	2.70	2.70	2.76	2.76

Table 14-24: Statistics of 2 m SG composites – capped vs uncapped

			Uncapped								Capped							
Domains		**Count**	**Mean**	**Std Dev**	**CV**	**Min**	**25th**	**50th**	**75th**	**Max**	**Mean**	**Std Dev**	**CV**	**Min**	**25th**	**50th**	**75th**	**Max**
Kemess South	**SG All**	**3,237**	**2.67**	**0.10**	**0.04**	**1.69**	**2.63**	**2.68**	**2.73**	**3.31**	**2.67**	**0.10**	**0.04**	**1.69**	**2.63**	**2.68**	**2.73**	**3.00**
	Asitka	109	2.70	0.11	0.04	2.48	2.62	2.70	2.75	3.16	2.70	0.10	0.04	2.48	2.62	2.70	2.75	3.00
	Takla	290	2.75	0.08	0.03	2.57	2.71	2.75	2.80	3.29	2.75	0.08	0.03	2.57	2.71	2.75	2.80	3.00
	Black Lake	2177	2.70	0.07	0.03	2.08	2.66	2.69	2.73	3.31	2.70	0.07	0.02	2.08	2.66	2.69	2.73	3.00
	Hazelton	631	2.55	0.09	0.03	1.69	2.51	2.56	2.60	2.92	2.55	0.09	0.03	1.69	2.51	2.56	2.60	2.92
	Epiclastic	21	2.65	0.11	0.04	2.42	2.61	2.65	2.72	2.91	2.65	0.11	0.04	2.42	2.61	2.65	2.72	2.91
	Overburden	9	2.71	0.03	0.01	2.68	2.70	2.70	2.72	2.76	2.71	0.03	0.01	2.68	2.70	2.70	2.72	2.76

Figure 14-24: Log probability plot of 2 m SG composites at Kemess South (a value of 3 was used to cap composites)



14.8 VARIOGRAPHY

Leapfrog Edge software was used to model variograms for gold, copper, and silver in the estimation domains. Experimental variograms and correlograms were generated for each mineralized domain, within individual structural domains. Downhole variograms were calculated to determine the nugget effect, which varies between 15% and 30% for the gold domains, 15% and 25% for the copper domains, and 20% and 25% for the silver domains of Kemess Main Zone. At Kemess South, the nugget effect varies 15–25% for gold domains, 15–30% for copper domains, and 10–40% silver domains.

Figure 14-25 and Figure 14-26 show example variogram models for gold for the low-grade gold domain at Kemess Main Zone and copper domain at Kemess South, and Table 14-25 and Table 14-26 provide summaries of the variogram parameters utilized in the final estimations.

Figure 14-25: Experimental variogram for the Nugget_KUG_0.2 gold domain at Kemess Main Zone

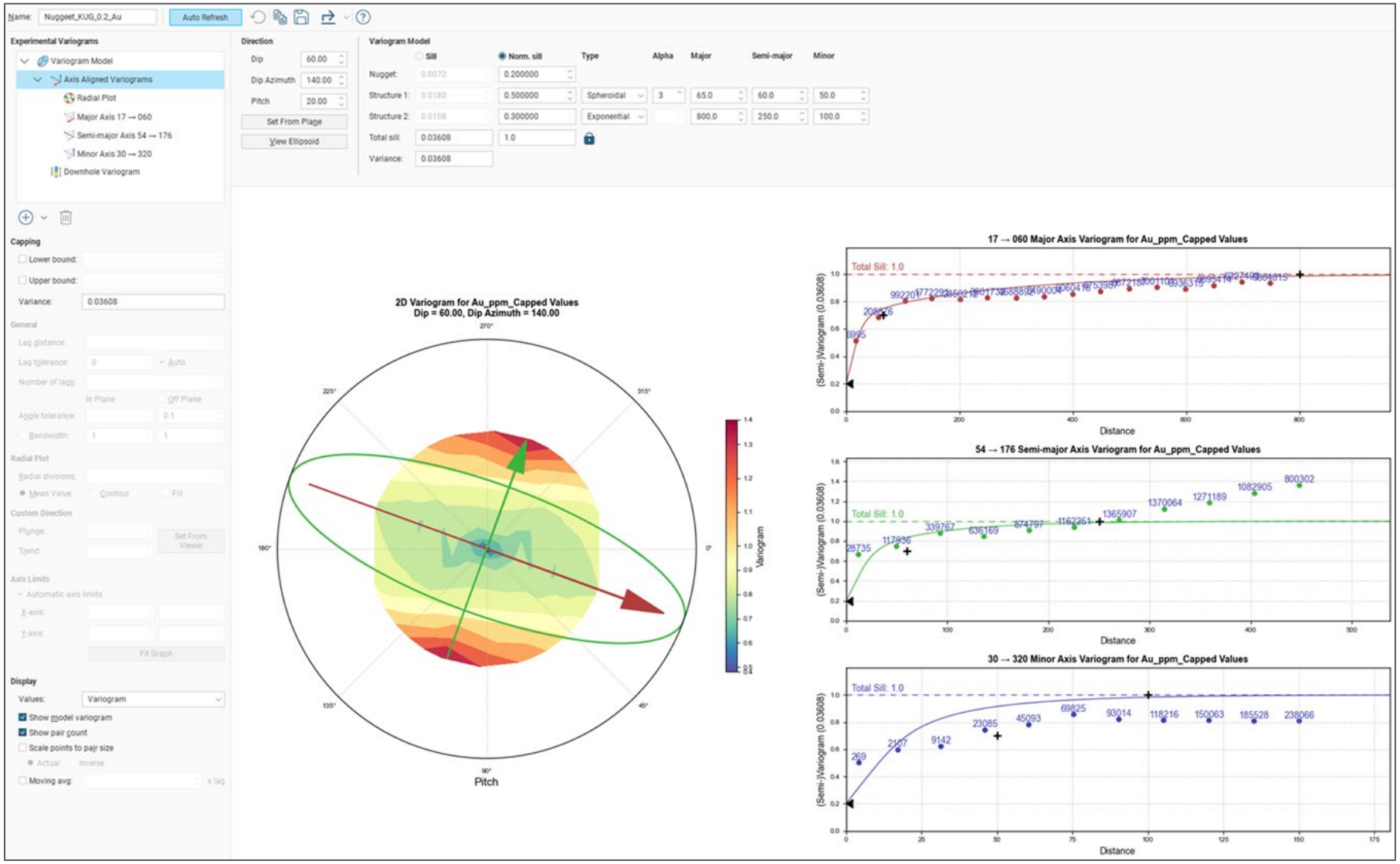


Figure 14-26: Experimental variogram for the copper domain in Fault Block 3 at Kemess South

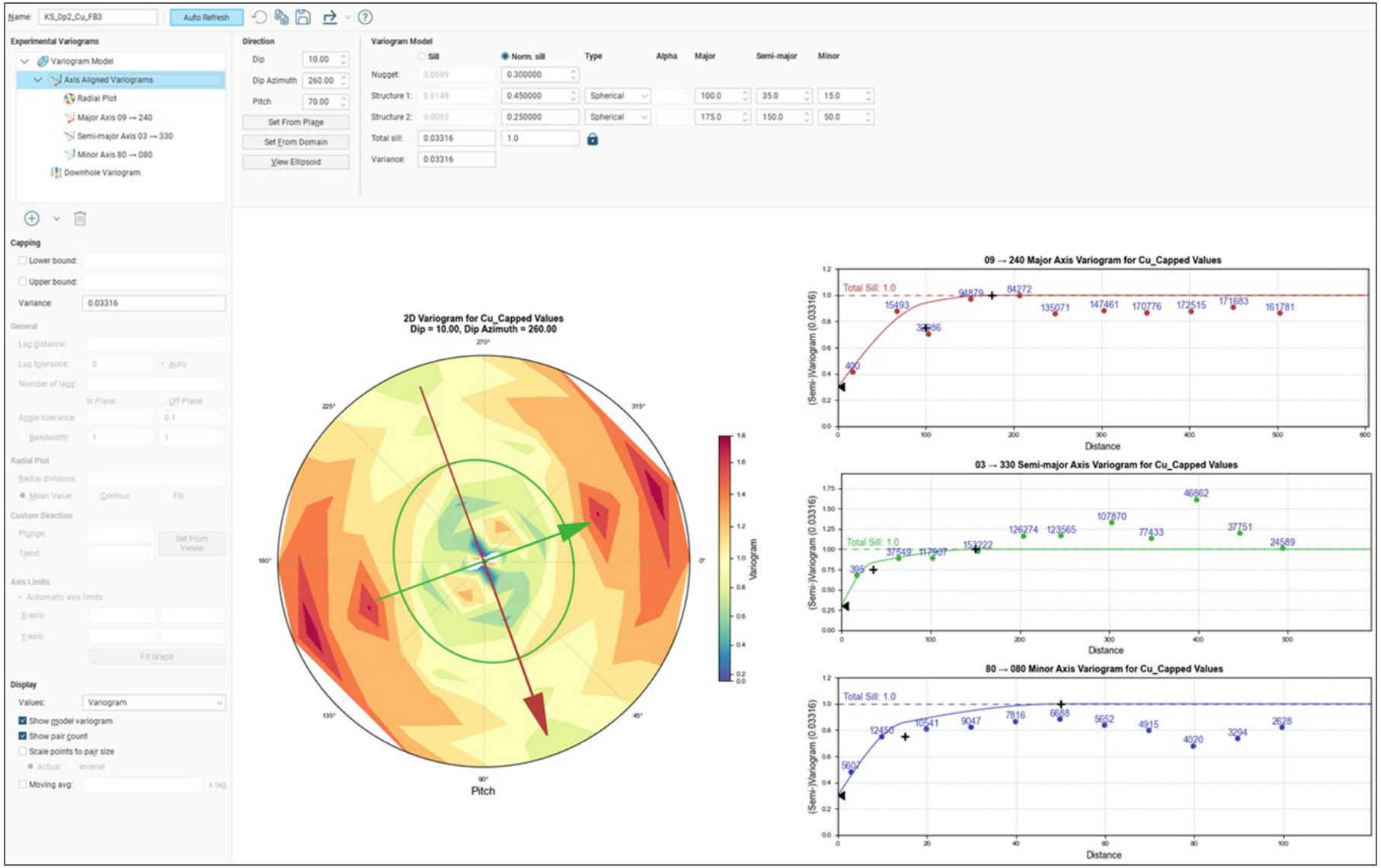


Table 14-25: Summary of variogram parameters used in the Kemess Main Zone MRE

Metal	Domain	Variography components												
		Direction			Nugget	Model type	First structure				Model type	Second structure			
		Dip	Dip az	Pitch			Sill	Range X (m)	Range Y (m)	Range Z (m)		Sill	Range X (m)	Range Y (m)	Range Z (m)
Au	Nugget_KUG 0.2	60	140	20	0.20	Spheroidal	0.50	65	60	50	Exponential	0.30	800	250	100
	KUG_0.6	60	140	25	0.30	Spherical	0.30	50	50	50	Spheroidal	0.40	230	130	80
	KE_0.3	60	140	25	0.15	Spheroidal	0.60	40	20	20	Spherical	0.25	200	200	200
	KE_0.6	60	140	20	0.25	Spherical	0.20	50	20	20	Spherical	0.55	70	60	60
Cu	Nugget_KUG 0.2	60	140	25	0.20	Spheroidal	0.55	35	35	25	Spheroidal	0.25	150	85	45
	KUG_0.4	60	140	20	0.20	Spherical	0.40	70	50	15	Spheroidal	0.40	110	60	50
	KE_0.2	60	140	20	0.15	Spheroidal	0.70	50	35	50	Spherical	0.15	400	400	200
	KE_0.4	60	140	20	0.25	Spheroidal	0.50	60	20	20	Spheroidal	0.25	250	160	130
Ag	KUG 1.5	60	140	60	0.20	Spherical	0.20	15	15	15	Spherical	0.60	85	80	15
	KE_0.2	60	140	70	0.25	Spheroidal	0.20	100	80	10	Spherical	0.55	190	190	10

Table 14-26: Summary of variogram parameters used in the Kemess South MRE

Metal	Domain	Variography components												
		Direction			Nugget	Model type	First structure				Model type	Second structure			
		Dip	Dip az	Pitch			Sill	Range X (m)	Range Y (m)	Range Z (m)		Sill	Range X (m)	Range Y (m)	Range Z (m)
Au	KS_0.2_Au_FB1	20	235	105	0.15	Spherical	0.2	100	100	30	Spherical	0.65	200	170	125
	KS_UNK_Au_FB1	20	235	110	0.15	Spherical	0.3	35	70	20	Spherical	0.55	200	200	200
	KS_UNK_Au_FB2	20	235	110	0.15	Spherical	0.3	35	70	20	Spherical	0.55	200	200	200
	KS_0.2_Au_FB3	20	235	105	0.15	Spherical	0.2	100	100	30	Spherical	0.65	200	170	125
	KS_UNK_Au_FB3	20	270	60	0.25	Spherical	0.35	100	200	50	Spherical	0.4	300	300	60
Cu	KS_UNK_Cu_FB1	10	10	260	0.15	Spherical	0.5	75	115	10	Spherical	0.35	200	180	50
	KS_UNK_Cu_FB2	10	10	260	0.15	Spherical	0.5	75	115	10	Spherical	0.35	200	180	50
	KS_0.2_Cu_FB3	15	10	260	0.3	Spherical	0.45	100	35	15	Spherical	0.25	175	150	50
	KS_UNK_Cu_FB3	30	10	260	0.15	Spherical	0.3	100	75	30	Spherical	0.55	200	150	50
Ag	KS_UKN_Ag_FB1	20	20	235	0.15	Spherical	0.3	35	70	20	Spherical	0.55	200	200	200
	KS_UNK_Ag_FB2	15	20	260	0.1	Spherical	0.35	180	150	15	Spherical	0.55	225	400	100
	KS_1.0_Ag_FB3	5	20	260	0.4	Spherical	0.1	75	60	5	Spherical	0.5	100	200	25
	KS_UNK_Ag_FB3	15	20	260	0.1	Spherical	0.35	180	150	15	Spherical	0.55	225	400	100

14.9 BLOCK MODEL

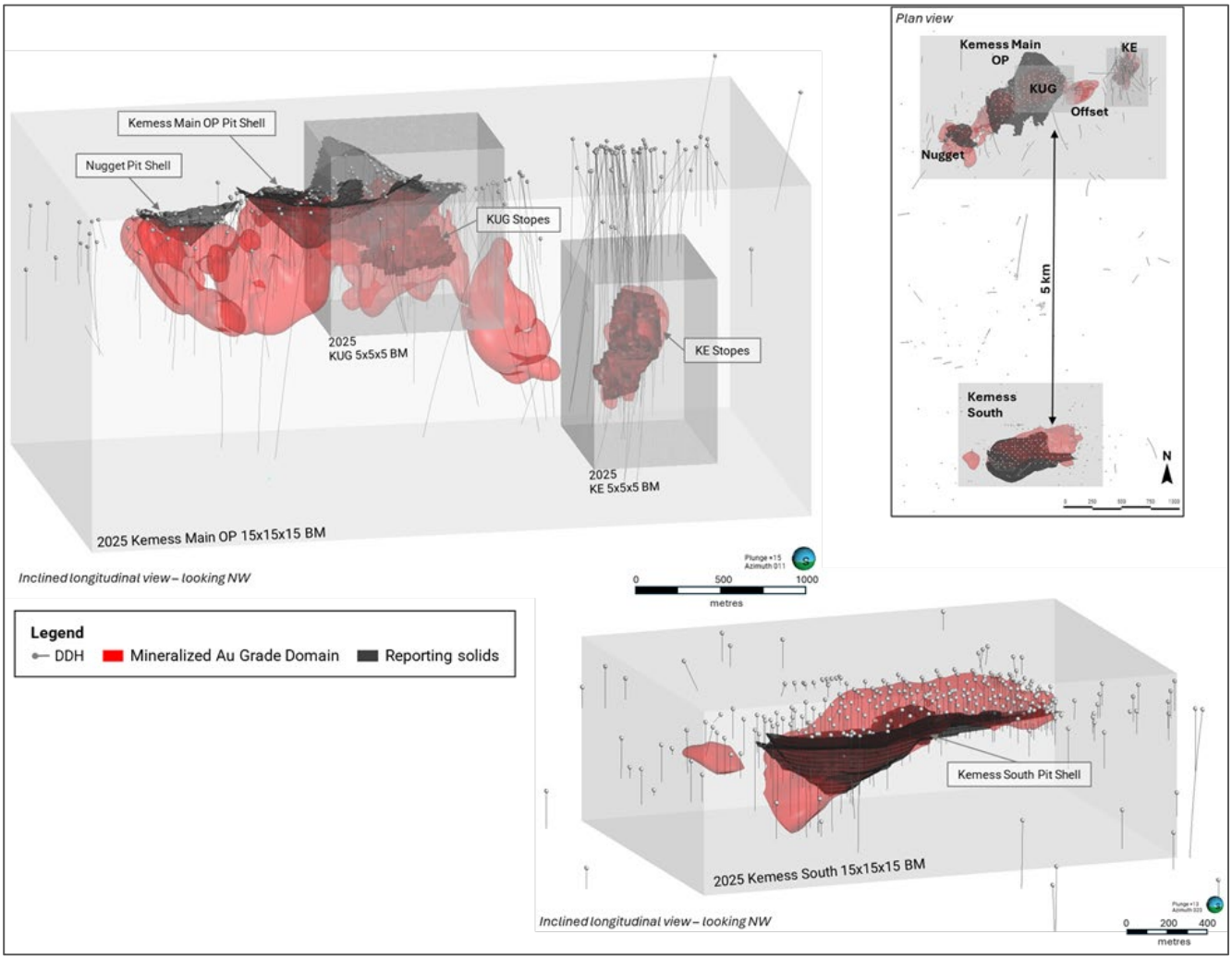
Three unrotated block models were created in Kemess Main zone: 1) Open Pit (Kemess Main Open Pit and Nugget), 2) KUG, and 3) KE using Seequent's Edge module within Leapfrog. The Open Pit block model has parent block size dimensions of 15 m x 15 m x 15 m (X, Y, and Z axis) which is consistent with planned open pit production bench heigh assumptions. KUG and KE block models have parent block size dimensions of 5 m x 5 m x 5 m (X, Y, and Z axis). The Open Pit model covers the entire Kemess Main zone resource area, and its block size is optimized for the evaluation of mineral resources considering an open pit mining scenario. The KUG and KE block models use smaller block sizes for the economic assessment of mineral resources when considering underground mining scenarios. While being entirely stand alone, the smaller block models use origins that place the smaller blocks fully inside the larger blocks of the Open Pit model, similar to a sub-blocked model. This approach was taken to ensure consistency across models, facilitate direct comparison of results, and allow for seamless integration of the different mining method scenarios. The underground portion of the deposit was drilled at relatively tight spacing (50–70 m) to support smaller block sizes typical of underground mining block models. Estimation parameters were adjusted to ensure results are comparable between the larger and smaller blocks. At Kemess South, one block model was created using the Seequent Edge module within Leapfrog. This block model has parent block size dimensions of 15 m x 15 m x 15 m (X, Y, and Z axis).

Table 14-27 summarizes the block model definitions. Figure 14-27 shows the spatial setting of the block models. The block model coordinates are based on the local UTM grid, NAD83 Zone 9N.

Table 14-27: Leapfrog Edge block model definition

Block model	Axis	Origin*	Block size (m)	Block count
Kemess Open Pit	X	634,200	15	286
	Y	6,324,850	15	166
	Z	2,000	15	134
KUG	X	635,850	5	207
	Y	6,326,005	5	165
	Z	2,000	5	219
Kemess East	X	637,440	5	147
	Y	6,326,110	5	204
	Z	1,115	5	225
Kemess South	X	634,865	15	168
	Y	6,319,500	15	120
	Z	1,515	15	63

Figure 14-27: Spatial setting of Kemess block models; KUG and Kemess East 5x5x5 m block centroids match those of the larger Kemess open pit 15x15x15 m block model



14.10 ESTIMATION PARAMETERS

Estimation parameters for gold, copper, and silver grades were based on variography studies on the capped composites. The metal grades across the block model were populated using ordinary kriging (OK). Grades were estimated in three passes with each pass using progressively larger search ellipsoids and increasingly relaxed data requirements (Table 14-28, Figure 14-28). Search distances were informed by the variogram ranges where the major, semi-major, and minor axis passes were set at 80, 90–100, and 95–250% of the total sill for the first, second, and third passes, respectively. The long, well-defined ranges as well as relatively low nugget in the variogram models support using a higher proportion of the modeled sill range for the first search pass. The subsequent passes reflect the anisotropy of the variogram model and ensure sufficient block coverage while remaining consistent with spatial continuity seen in the data.

At Kemess Main Zone, a one-way soft boundary of 15 m (one block size) was applied for the low-grade domain whereby low-grade domains were informed by composites 15 m beyond the modelled boundary. Allowing limited cross-boundary influence recognizes geological continuity and gradational contact between the high-grade and low-grade domains while preserving the integrity of the high-grade domain by preventing dilution from the surrounding lower grade material. Visual validation stepping through cross sections of the block model against composites as well as validation through swath plots confirmed that the 15 m one-way soft boundary provides reasonable grade distribution (Figure 14-28).

Figure 14-28: Swath plot validation of KUG low grade domain displaying both hard and soft boundary capped gold composites against block model parallel estimates

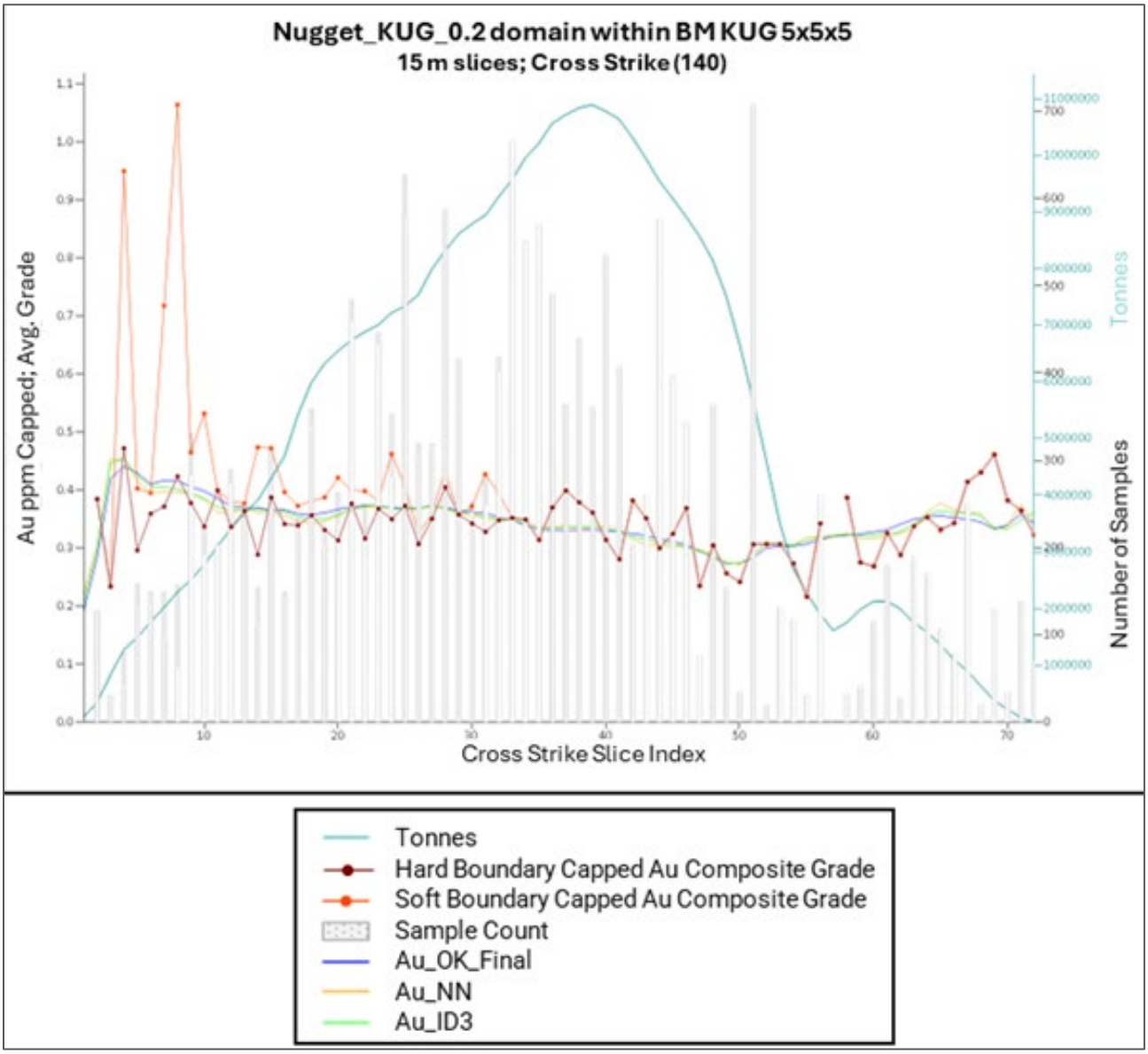


Table 14-28: Data and search parameters for Kemess Main Zone estimation

Metal	BM	Domain	Ellipsoid directions			Pass 1							Pass 2							Pass 3						
			Dip	Dip az.	Pitch	Range (m)			Comps		Max samples per hole	Min DDH	Range (m)			Comps		Max samples per hole	Min DDH	Range			Comps		Max samples per hole	Min DDH
						X	Y	Z	Min	Max			X	Y	Z	Min	Max			X	Y	Z	Min	Max		
Au	Open pit (15 x 15 x 15 m)	Nugget_KUG_0.2	60	140	20	80	60	40	10	16	4	3	160	120	80	8	20	4	2	350	225	150	6	24	-	1
		Unmineralized	60	140	20	50	50	50	10	16	4	3	100	100	100	8	20	4	2	300	300	300	6	24	-	1
	KUG (5 x 5 x 5 m)	Nugget_KUG_0.2	60	140	20	80	60	40	5	8	2	3	160	120	80	4	10	2	2	350	225	150	3	12	-	1
		KUG_0.6	60	140	25	60	30	20	5	8	2	3	90	50	30	4	10	2	2	230	130	80	3	12	-	1
		Unmineralized	60	140	20	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1
	KE (5 x 5 x 5 m)	KE_0.3	60	140	25	50	50	50	5	8	2	3	75	75	75	4	10	2	2	200	200	200	3	12	-	1
		KE_0.6	60	140	20	45	40	40	5	8	2	3	70	60	60	4	10	2	2	250	150	150	3	12	-	1
		Unmineralized	60	140	20	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1
Cu	Open pit (15 x 15 x 15 m)	Nugget_KUG_0.2	60	140	25	80	60	40	10	16	4	3	160	120	80	8	20	4	2	325	225	150	6	24	-	1
		Unmineralized	60	140	20	50	50	50	10	16	4	3	100	100	100	8	20	4	2	300	300	300	6	24	-	1
	KUG (5 x 5 x 5 m)	Nugget_KUG_0.2	60	140	25	80	60	40	5	8	2	3	160	120	80	4	10	2	2	325	225	150	3	12	-	1
		KUG_0.4	60	140	20	45	25	20	5	8	2	3	65	35	30	4	10	2	2	110	60	50	3	12	-	1
		Unmineralized	60	140	20	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1
	KE (5 x 5 x 5 m)	KE_0.2	60	140	20	40	40	20	5	8	2	3	80	80	40	4	10	2	2	325	325	100	3	12	-	1
		KE_0.4	60	140	20	50	30	25	5	8	2	3	75	50	40	4	10	2	2	250	160	130	3	12	-	1
		Unmineralized	60	140	20	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1
Ag	Open pit (15 x 15 x 15 m)	KE_2.0	60	140	70	60	60	30	10	16	4	3	95	95	45	8	20	4	2	190	190	95	6	24	-	1
		Unmineralized	60	140	70	50	50	50	10	16	4	3	100	100	100	8	20	4	2	300	300	300	6	24	-	1
	KUG (5 x 5 x 5 m)	KUG_1.5	60	140	60	50	45	20	5	8	2	3	85	80	40	4	10	2	2	210	200	100	3	12	-	1
		Unmineralized	60	140	70	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1
		Offset_1.5	60	140	70	65	55	25	5	8	2	3	105	90	45	4	10	2	2	210	180	90	3	12	-	1
	KE (5 x 5 x 5 m)	KE_2.0	60	140	70	60	60	30	5	8	2	3	95	95	45	4	10	2	2	190	190	95	3	12	-	1
		Unmineralized	60	140	70	50	50	50	5	8	2	3	100	100	100	4	10	2	2	300	300	300	3	12	-	1

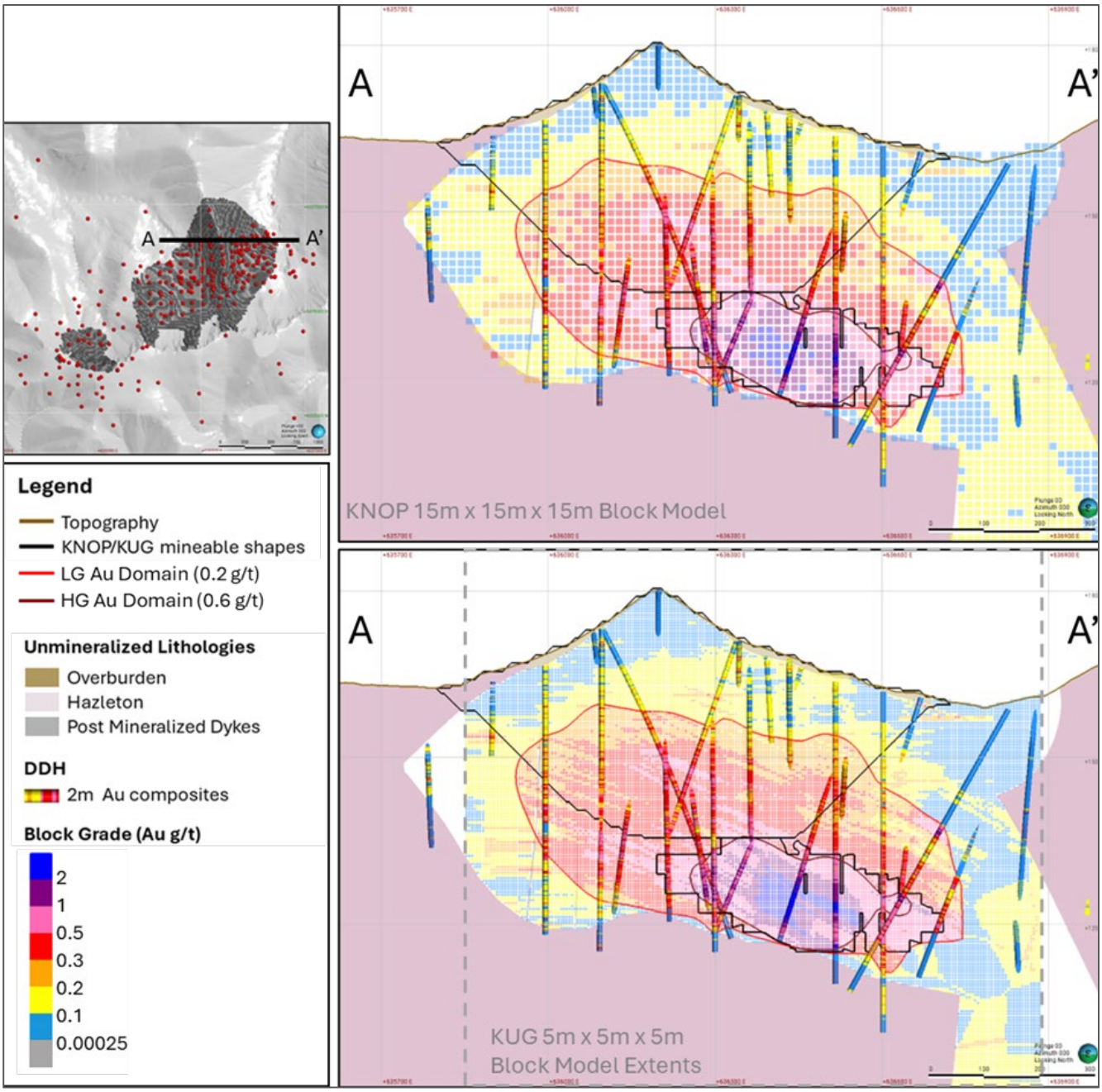
Table 14-29: Data and search parameters for Kemess South estimation

Metal	BM	Domain	Ellipsoid directions			Pass 1							Pass 2							Pass 3						
			Dip	Dip az.	Pitch	Range (m)			Comps		Max samples per hole	Min DDH	Range (m)			Comps		Max samples per hole	Min DDH	Range			Comps		Max samples per hole	Min DDH
						X	Y	Z	Min	Max			X	Y	Z	Min	Max			X	Y	Z	Min	Max		
Au	Kemess South (15 x 15 x 15 m)	KS_0.2_Au_FB1	variable			100	85	62.5	10	16	4	3	200	170	125	8	20	4	2	300	255	180	6	24	-	1
		KS_UNK_Au_FB1	variable			75	75	15	10	16	4	3	150	150	30	8	20	4	2	300	300	60	6	24	-	1
		KS_UNK_Au_FB2	20	270	60	75	75	15	10	16	4	3	150	150	30	8	20	4	2	300	300	60	6	24	-	1
		KS_0.2_Au_FB3	variable			100	85	62.5	10	16	4	3	200	170	125	8	20	4	2	300	255	180	6	24	-	1
		KS_UNK_Au_FB3	variable			75	75	15	10	16	4	3	150	150	30	8	20	4	2	300	300	60	6	24	-	1
Cu		KS_UNK_Cu_FB1	variable			50	50	50	10	16	4	3	75	75	75	8	20	4	2	200	200	200	6	24	-	1
		KS_UNK_Cu_FB2	10	260	75	45	40	40	10	16	4	3	70	60	60	8	20	4	2	250	150	150	6	24	-	1
		KS_0.2_Cu_FB3	variable			50	50	50	10	16	4	3	100	100	100	8	20	4	2	300	300	300	6	24	-	1
		KS_UNK_Cu_FB3	variable			80	60	40	10	16	4	3	160	120	80	8	20	4	2	325	225	150	6	24	-	1
Ag		KS_UKN_Ag_FB1	variable			100	75	25	10	16	4	3	200	150	50	8	20	4	2	300	225	75	6	24	-	1
		KS_UNK_Ag_FB2	20	270	60	100	75	25	10	16	4	3	200	150	50	8	20	4	2	300	225	75	6	24	-	1
		KS_1.0_Ag_FB3	variable			87.5	75	25	10	16	4	3	175	150	50	8	20	4	2	350	300	100	6	24	-	1
		KS_UNK_Ag_FB3	variable			100	75	25	10	16	4	3	200	150	50	8	20	4	2	300	225	75	6	24	-	1

Finally, overburden and post-mineralized units (post mineralized dykes and the Hazelton Formation) are barren and thus do not have metal content within the block models. Block grades in these domains were assigned zero grades.

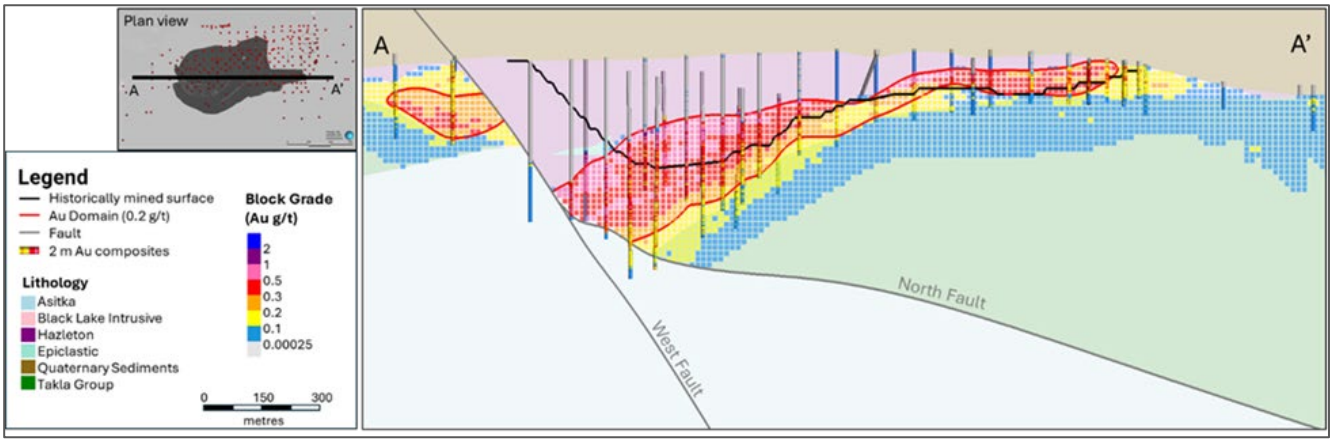
Figure 14-29 and Figure 14-30 show the grade distribution along a typical east-west section of the Nugget, Kemess Main open pit, and KUG deposits and Kemess South deposit respectively.

Figure 14-29: East-west cross section through the Kemess Main open pit conceptual pit shell and KUG MSOs



Note the good agreement between 2 m composites and block grades; top image displays the 15x15x15 m Kemess Main Open Pit block model; lower image displays the 5x5x5 m KUG block model.

Figure 14-30: East-west cross section through the Kemess South block model area showing good agreement between 2 m composites and block grades



14.11 BLOCK MODEL VALIDATION

The validation of the block model estimate was carried out through a multifaceted approach, utilizing visual, comparative, and statistical methods:

- **Visual inspection**: Visual examinations were conducted on sections and plans to ensure good agreement between informing composites and block grades. This step was crucial to confirm that the block grades were accurately represented and to ensure that instances of high-grade samples did not lead to unrealistically large volumes of high-grade blocks in the model. Mineralization trends in drillholes were also compared to the distribution of grade blocks in the model.
- **Comparative analysis**: To assess the robustness of the MRE, a series of parallel estimation scenarios were completed. Table 14-30, Table 14-31, and Table 14-32 show the results of the comparative inventory models – Inversed Distance (ID) and NN – within the conceptual pit shell and MSOs generated for Mineral Resource reporting. Open pit analyses were run on gold cutoffs at 0.22 g/t Au based cut-off for ease and consistency of reporting in the Leapfrog software. This gold cut-off was selected as it is broadly equivalent to the NSR cut-off applied for Mineral Resource reporting and provides a comparable basis for evaluating open pit inventory. Underground inventory analysis was completed within the MSOs generated at NSR cutoffs of CA$54.10/t for KUG and Kemess East and include internal dilution.

Kemess Main Zone parallel estimates agree well between the estimates using alternative estimators suggesting that the OK estimator provides a reasonable global estimate of the metal content in the deposit.

Parallel estimates for the Kemess South deposit show significant discrepancies between the NN estimates when compared to OK. The discrepancy between NN (using 2 m composites) and OK indicates strong short-range variability that are characteristic of a high nugget effect. NN estimates using 6 m composites were compared to the 2 m OK estimates for all metals. The increased composite length in the NN estimate is smoother and aligns better with OK, implying underlying mineralization is more continuous at scales larger than 2 m and short-scale variability is not representative of block support. The discrepancies between NN and OK within the Kemess South inferred inventory is more pronounced due to the low tonnages, but such differences are not considered significant given the small tonnage represented.

Note: For the following three tables, Model Inventory is reported at an NSR cut-off value of CA$15.97/t for Kemess Main Open Pit and Kemess South, and CA$54.10/t for KUG and Kemess East. Reporting within the conceptual pit shells and MSOs were used. The MRE was reported using Deswik software, while validation was completed in Leapfrog Edge software. This may result in minor discrepancies between table values.

Table 14-30: Gold value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates

Deposit	Model	Indicated						Inferred					
		Mass ('000 t)	% Diff. to OK model	Au grade (g/t)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model	Mass ('000 t)	% Diff. to OK model	Au grade (g/t)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model
Kemess Main Open Pit	OK	131,502	-	0.32	-	1,347	-	92,447	-	0.30	-	902	-
	ID3	130,116	-1.06%	0.32	0.98%	1,346	-0.08%	91,958	-0.53%	0.31	1.01%	906	0.48%
	NN	122,747	-6.89%	0.34	6.16%	1,337	-0.73%	87,997	-4.93%	0.32	5.27%	905	0.34%
	Metal Loss	132,073	0.43%	0.32	0.00%	1,353	0.43%	92,201	-0.27%	0.30	0.42%	904	0.16%
KUG	OK	34,274	-	0.86	-	943	-	20,729	-	0.78	-	517	-
	ID3			0.85	-0.58%	938	-0.58%				2.32%	529	2.32%
	NN			0.85	-1.02%	934	-1.02%				3.14%	534	3.14%
	Metal Loss			0.86	0.61%	949	0.61%				0.49%	520	0.49%
Kemess East	OK	37,328	-	0.67	-	608	-	37,328	-	0.59		830	
	ID3			0.66	-1.11%	601	-1.11%				-1.31%	820	-1.31%
	NN			0.66	-0.98%	602	-0.98%				-1.45%	818	-1.45%
	Metal Loss			0.67	0.39%	610	0.39%				0.59%	835	0.59%
Kemess South	OK	13,187	-	0.38	-	163	-	179	-	0.34	-	1.9	-
	ID3	13,003	-1.40%	0.39	0.15%	161	-1.25%	179	0.00%	0.32	-4.25%	1.9	-4.25%
	NN	11,625	-12.59%	0.40	5.04%	151	-7.57%	73	-83.84%	0.35	5.05%	0.8	-79.63%
	NN_6m	13,312	0.94%	0.38	-2.38%	166	1.81%	179	0.00%	0.28	-19.35%	1.6	-19.35%
	Metal Loss	13,187	0.00%	0.38	0.00%	163	0.00%	179	0.00%	0.34	0.00%	1.9	0.00%

Table 14-31: Copper value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates

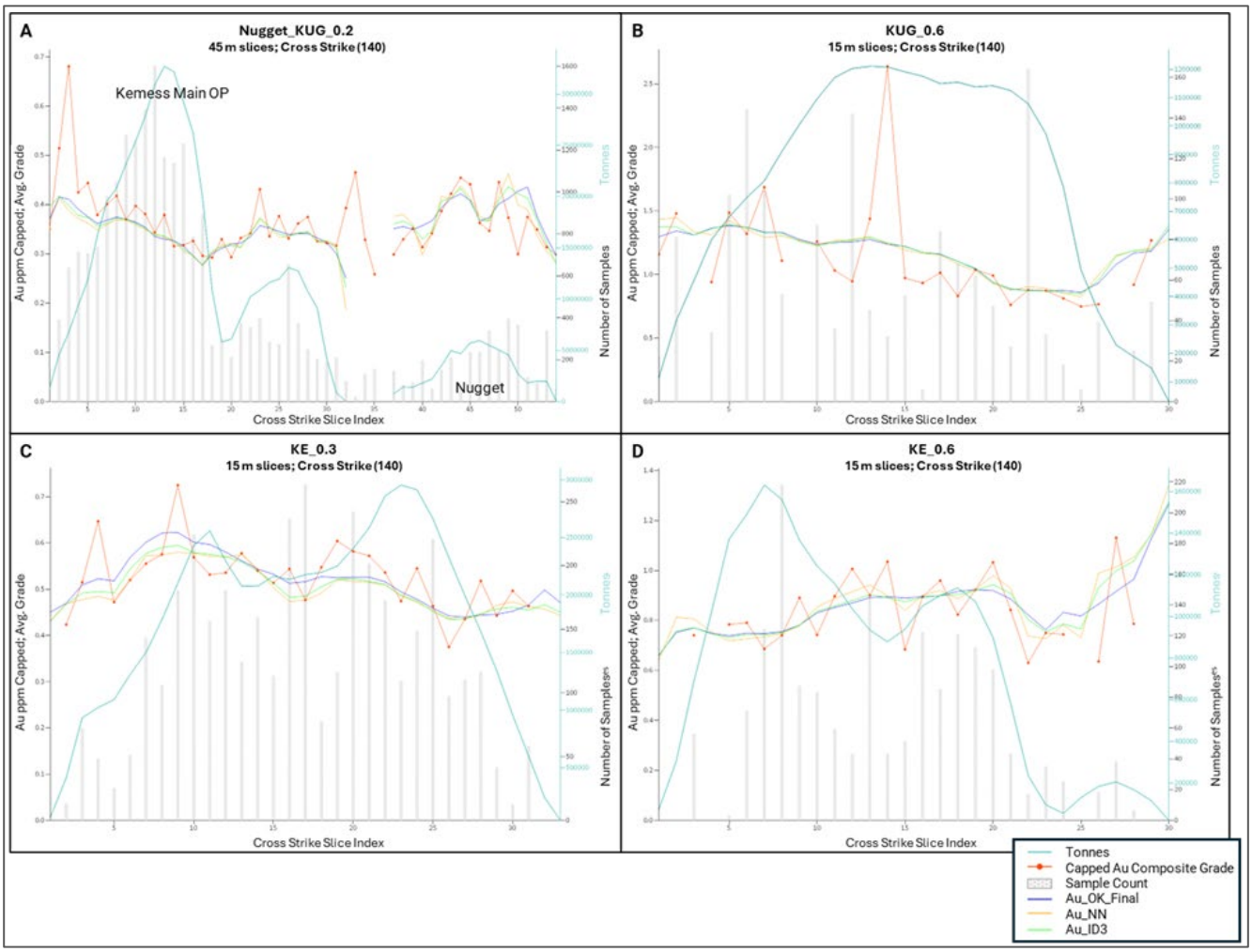
Deposit	Model	Indicated						Inferred					
		Mass ('000 t)	% Diff. to OK model	Cu grade (%)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model	Mass ('000 t)	% Diff. to OK model	Cu grade (%)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model
Kemess Main Open Pit	OK	131,502	-	0.16	-	450	-	92,447	-	0.14	-	288	-
	ID3	130,116	-1.06%	0.16	1.25%	451	0.19%	91,958	-0.53%	0.14	2.07%	292	1.53%
	NN	122,747	-6.89%	0.16	3.21%	434	-3.67%	87,997	-4.93%	0.15	5.05%	288	0.11%
	Metal Loss	132,073	0.43%	0.16	0.49%	454	0.93%	92,201	-0.27%	0.14	0.88%	289	0.61%
KUG	OK	34,274	-	0.38	-	285	-	65,800	-	0.35	-	159	-
	ID3			0.38	-0.58%	284	-0.58%				1.63%	161	1.63%
	NN			0.37	-0.91%	283	-0.91%				2.01%	162	2.01%
	Metal Loss			0.39	3.06%	292	3.06%				0.64%	162	0.64%
Kemess East	OK	28,255	-	0.46	-	288	-	43,426	-	0.44	-	423	-
	ID3			0.46	0.00%	288	0.00%				0.29%	424	0.29%
	NN			0.46	-0.78%	286	-0.78%				-0.69%	420	-0.69%
	Metal Loss			0.46	0.56%	289	0.56%				0.55%	426	0.55%
Kemess South	OK	13,187	-	0.13	-	38	-	179	-	0.08	-	0.3	-
	ID3	13,003	-1.40%	0.13	-0.24%	37	-1.64%	179	0.00%	0.08	-1.93%	0.3	-1.93%
	NN	11,625	-12.59%	0.13	0.29%	33	-12.30%	73	-83.84%	0.04	-64.67%	0.1	-130.79%
	NN_6m cmp	13,312	0.94%	0.13	0.67%	38	1.62%	179	0.00%	0.07	-6.35%	0.3	-6.35%
	Metal Loss	13,187	0.00%	0.13	0.59%	38	0.59%	179	0.00%	0.08	0.00%	0.3	0.00%

Table 14-32: Silver value comparison parallel estimates of NN, ID, and uncapped model (metal loss) to OK estimates

Deposit	Model	Indicated						Inferred					
		Mass ('000 t)	% Diff. to OK model	Ag grade (g/t)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model	Mass ('000 t)	% Diff. to OK model	Ag grade (g/t)	% Diff. to OK model	Metal content ('000 t.oz)	% Diff. to OK model
Kemess Main Open Pit	OK	131,502	-	1.12	-	4,737	-	92,447	-	1.01	-	3,002	-
	ID3	130,116	-1.06%	1.13	1.00%	4,735	-0.06%	91,958	-0.53%	1.03	2.30%	3,056	1.76%
	NN	122,747	-6.89%	1.15	2.95%	4,554	-3.94%	87,997	-4.93%	1.04	2.96%	2,944	-1.97%
	Metal Loss	132,073	0.43%	1.12	-0.71%	4,772	0.78%	92,201	-0.27%	1.04	3.13%	3,090	2.87%
KUG	OK	34,274	-	2.59	-	2,851	-	20,729	-	2.31	-	1,541	-
	ID3			2.60	-0.52%	2,866	0.52%			2.33	0.77%	1,553	0.77%
	NN			2.61	-0.68%	2,871	0.68%			2.34	1.19%	1,560	1.19%
	Metal Loss			2.65	2.27%	2,917	2.27%			2.37	2.42%	1,579	2.42%
Kemess East	OK	28,255	-	1.99	-	1,811	-	43,426	-	2.00	-	2,797	-
	ID3			2.01	1.00%	1,829	1.00%			2.06	2.62%	2,872	2.62%
	NN			2.03	1.62%	1,841	1.62%			2.07	3.37%	2,893	3.37%
	Metal Loss			2.01	0.86%	1,827	0.86%			2.04	1.79%	2,848	1.79%
Kemess South	OK	13,187	-	0.69	-	291	-	179	-	0.39	-	2.2	-
	ID3	13,003	-1.40%	0.68	-0.99%	284	-2.39%	179	0.00%	0.40	2.80%	2.3	2.80%
	NN	11,625	-12.59%	0.69	0.23%	257	-12.37%	73	-83.84%	0.50	25.27%	1.2	-61.85%
	NN_6m	13,312	-0.94%	0.66	-4.16%	282	-3.22%	179	0.00%	0.35	-11.27%	2.0	-11.27%
	Metal Loss	13,187	0.00%	0.70	1.80%	296	1.80%	179	0.00%	0.39	0.00%	2.2	0.00%

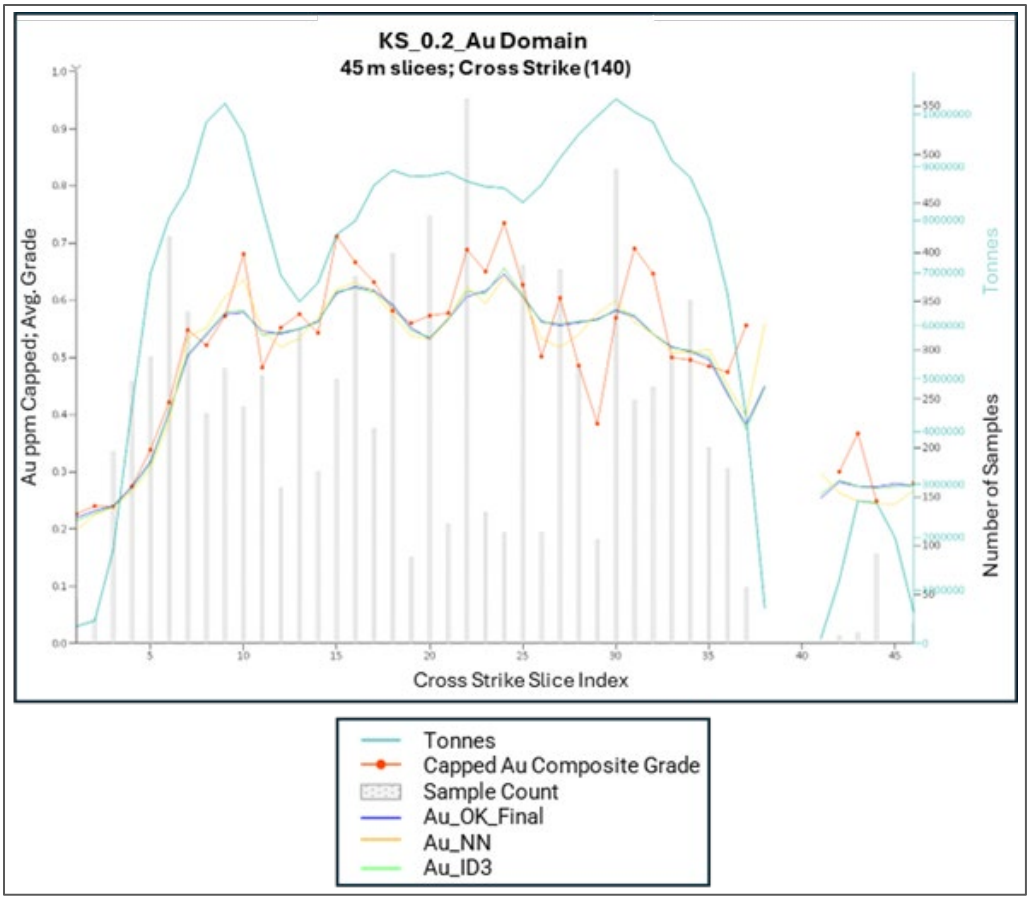
Swath plots were prepared to assess the compliance of the estimates to the informing data; they show good correlation between the three estimators as well as the informing composites and are shown in Figure 14-31 and Figure 14-32.

Figure 14-31: Cross strike gold swath plots through the Kemess Main Zone deposits



Note: A: Nugget_KUG_0.2 Domain, 15x15x15 m block model; B: KUG_0.6 Domain, 5x5x5 m KUG block model; C: KE_0.3 Domain, 5x5x5 m Kemess East block model; D: KE_0.6 Domain, 5x5x5 m Kemess East block model.

Figure 14-32: Cross strike gold swath plot of the mineralized gold domain at Kemess South



14.12 CLASSIFICATION

Mineral resource classification is a subjective concept, and industry best practices suggest that a mineral resource classification should consider the confidence in the geological continuity of the mineralization domains, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim to integrate all these concepts to delineate regular areas of similar resource classification.

Block model quantities and grade estimates for the Kemess Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Karen Chiu, PGeo (PGO#2753), and reviewed by Dr. Lars Weiershäuser, PGeo, (PGO#1504), Director of Geology for Centerra.

The QP is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. Sampling information was acquired by core drilling.

The process of classification for this project was executed in two main stages:

- Initial coding: In the first stage, blocks were assigned codes based on specific parameters detailed in Table 14-33. This stage establishes a preliminary classification based on quantifiable data.
- Manual smoothing (Figure 14-33 and Figure 14-34): The second stage involved a manual smoothing process. This step was crucial in addressing isolated instances where small clusters of blocks were assigned a classification level that significantly differed from their surrounding blocks. To achieve a more coherent classification, “classification solids” were created. These solids provided a basis for re-coding the block model, thereby ensuring a more uniform final classification.

While initial coding classified blocks as measured at Kemess Main Zone, the drillhole spacing was insufficient to establish continuity, resulting in a discontinuous distribution of pattern, commonly referred to as “spotted-dog distribution”. The highest level of classification at both projects is indicated.

Kemess South classification at indicated and inferred uses wider drill hole spacing criteria than Kemess Main Zone. These criteria are consistent with historical parameters applied while the mine was in production and remain appropriate given the established production history at Kemess South.

Table 14-33: Parameters for classification

Class	Parameters	
	Kemess Main Zone	**Kemess South**
Measured	• <50 m average drill hole spacing • 3 or more holes in grade estimation • Manual smoothing	• <50 m average drill hole spacing • 3 or more holes in grade estimation • Manual smoothing
Indicated	• <70 m average drill hole spacing • 2 or more holes in grade estimation • Manual smoothing	• <100 m average drill hole spacing • 2 or more holes in grade estimation • Manual smoothing
Inferred	• <100 m average drill hole spacing • Manual smoothing	• <150 m average drill hole spacing • Manual smoothing

Figure 14-33: Oblique view of Kemess Main Zone 15x15x15 classified block model; pre and post manual smoothing

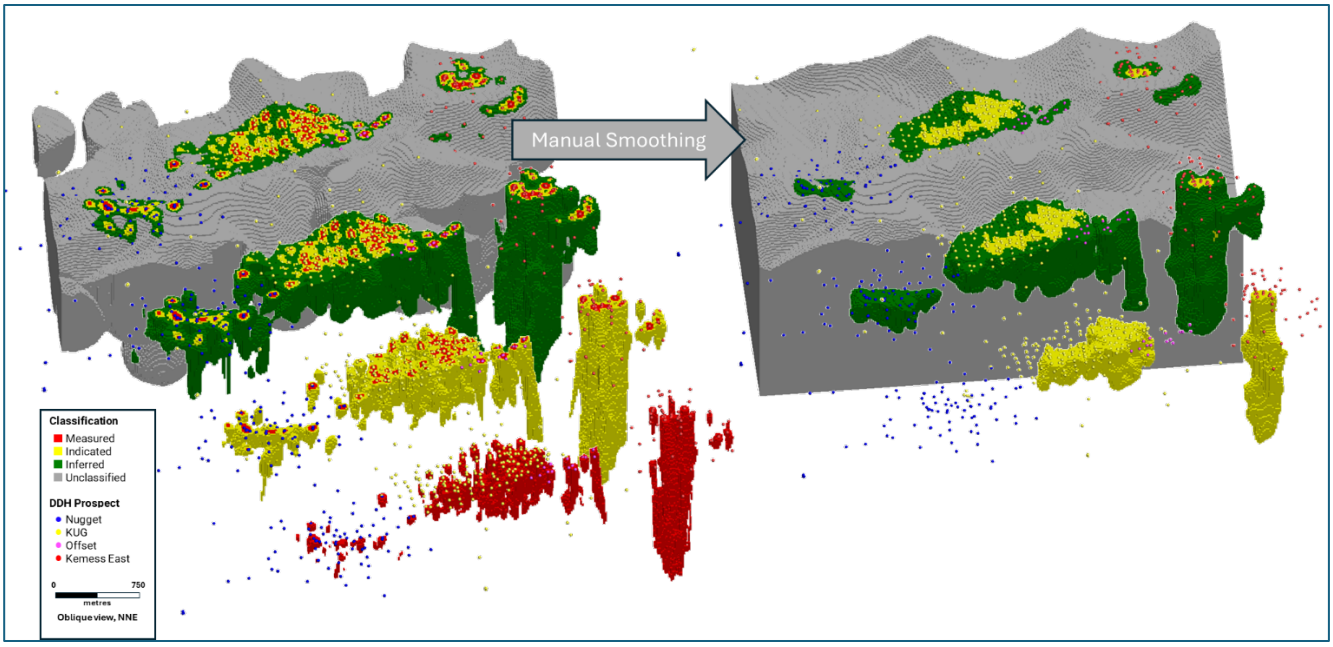
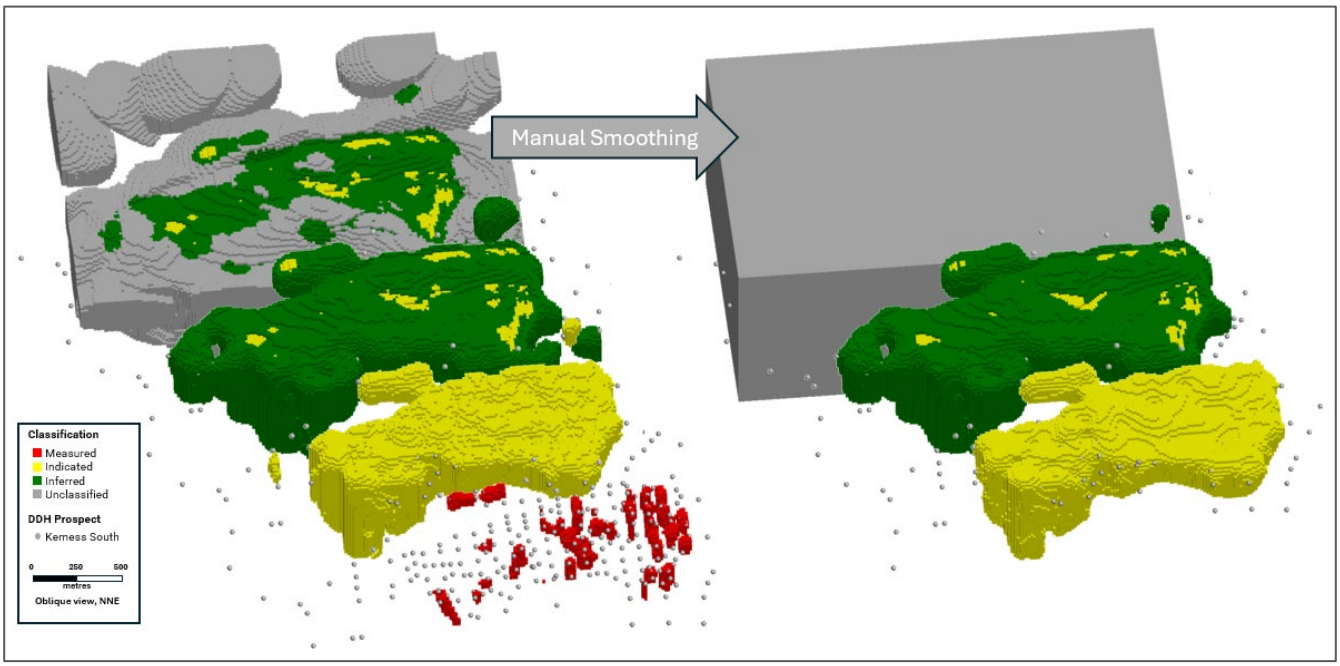


Figure 14-34: Oblique view of Kemess South 15x15x15 classified block model; pre and post manual smoothing



14.13 MINERAL RESOURCE STATEMENT

CIM Definition Standard for Mineral Resources and Mineral Reserves (CIM, May 2014) define a Mineral Resource 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."

The "reasonable prospects for eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery. Centerra considers that the gold, copper, and silver mineralization of the Kemess Main Zone and Nugget deposits is amenable to open pit extraction, and that of the KUG and Kemess East deposits is amenable to underground extraction. A pit optimizer was used to assist with determining which portions of the deposits show a "reasonable prospect for eventual economic extraction" from an open pit and to assist with selecting reporting assumptions.

Table 14-34: Optimization parameters used to generate conceptual Kemess pits and underground mineable shapes

Parameter / Cost Centre	Units	Kemess Main Zone	Kemess South	KUG	Kemess East
Mining method		Open pit	Open pit	35 m LHS	35 m LHS
OSA range (OP only)	degrees	30-45	22-50		
Mining costs					
Open pit	CA$/t mined	3.50	6.00	-	-
Incremental open pit mining	CA$/t/bench mined	0.06	0.06	-	-
Talus Zone mining	CA$/t mined	0.60	-	-	-
Underground direct stoping	CA$/t mined	25.00			
Underground backfill	CA$/t mined	9.00			
Underground grade control	CA$/t mined	1.50			
Underground development	CA$/t mined	2.00			
Processing	CA$/t mined	9.60			
Ore conveyance	CA$/t milled	0.70			
G&A	CA$/t milled	3.67			
TSF	CA$/t milled		2.00		
Transportation	US$/WMT	200.39			
Sustaining capital	CA$/t milled	2.00		4.00	
Smelter charges					
Unit smelting charge	US$/t milled	75.00			
Gold refining	US$/t milled	5.00			
Copper refining	US$/t milled	0.07			
Concentrate copper grade	%	21			
Concentrate loss factor	%	99.5			
Concentrate moisture content	%	8			

Parameter / Cost Centre	Units	Kemess Main Zone	Kemess South	KUG	Kemess East
Recoveries before leaching					
Au / Au in Broken Zone	%	60.3 / 53.3	67.0	AuR = 7.3536x + 63.261 where x is gold head grade (max 84.3%)	AuR = 7.3536x + 63.261 where x is gold head grade (max 84.3%)
Cu / Cu in Broken Zone	%	91.1 / 83.2	85.0	CuR = 101.26$x^{0.0733}$ where x is gold head grade (max 95%)	CuR = 101.26$x^{0.0733}$ where x is gold head grade (max 96%)
Au in Nugget area	%	63.4		-	-
Cu in Nugget area	%	90.0		-	-
Additional recovery from leach circuit					
Au / Au in Broken Zone	%	+14 / 18	+6	+14 (max recovery applies)	+14 (max recovery applies)
Gold price (US$/oz)		2,400			
Copper price (US$/lb)		4.00			
Silver price (US$/oz)		25.00			
Royalty (%)		0			
Exchange rate (CA$/US$)		1.33			

The cost and parameter assumptions for this resource estimate were largely based on prior Centerra work for the Kemess Project, with open pit mining costs and additional costs such as bench mining, transportation, smelter charges, and G&A were benchmarked against similar operations and existing agreements and adjusted to reflect the preliminary nature of the estimate. Underground mining costs were verified by independent consultants based on similar operations using longhole stoping and paste backfill. Key equipment assumptions include 140-t haul trucks and hydraulic excavators under an owner-operated model, with higher costs applied to the talus zone due to expected mining difficulty.

The QP considers that it is appropriate to report the mineral resources for the Kemess Main Open Pit at a NSR cut-off value of CA$15.97/t. Mineral resources are reported within a constraining pit shell and take into consideration metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges. Underground resources are reported with an NSR cut-off value of CA$54.10/t that takes into consideration metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges. Mineral resources were estimated in conformity with the generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines.

The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The mineral resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-

economic, and other factors. The Mineral Resource Statement for the Kemess deposits is presented in Table 14-35. The effective date of the Mineral Resource Statement is December 31, 2025.

Table 14-35: Updated Kemess Resource Summary[1,4,5], as of December 31, 2025

Indicated Mineral Resources[2]							
	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)	Copper Grade (%)	Contained Copper (Mlbs)	Silver Grade (g/t)	Contained Silver (koz)
Open Pit - Main Zone	170,513	0.30	1,668	0.15	575	1.12	6,155
Open Pit - Kemess South	13,204	0.37	158	0.13	38	0.68	289
Underground - Kemess UG	33,223	0.82	877	0.36	265	2.48	2,652
Underground - Kemess East	27,491	0.64	565	0.44	268	1.91	1,684
Total Indicated	**244,431**	**0.42**	**3,269**	**0.21**	**1,146**	**1.37**	**10,780**
Inferred Mineral Resources[3]							
	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)	Copper Grade (%)	Contained Copper (Mlbs)	Silver Grade (g/t)	Contained Silver (koz)
Open Pit - Main Zone	237,050	0.30	2,299	0.13	682	1.06	8,108
Open Pit - Kemess South	198	0.34	2	0.08	0	0.42	3
Underground - Kemess UG	20,094	0.74	481	0.33	148	2.22	1,433
Underground - Kemess East	42,252	0.57	772	0.42	393	1.92	2,602
Total Inferred	**299,593**	**0.37**	**3,555**	**0.19**	**1,223**	**1.26**	**12,146**

Notes:

1. Mineral resources are stated in accordance with CIM (2014) Definitions as incorporated by reference into NI 43-101. Mineral Resources are estimated and have an effective date of December 31, 2025.
2. Mineral resources do not have demonstrated economic viability.
3. Inferred mineral resources have a lower level of confidence as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the inferred mineral resources will ever be upgraded to a higher category.
4. Centerra's equity interests are as follows: Kemess Main, Kemess South, KUG, Kemess East 100%.
5. Numbers may not sum precisely due to rounding.

Additional Resource footnotes:

- A conversion factor of 31.1035 grams per troy ounce of gold is used in the mineral reserve and resource estimates.
- The mineral resources are reported based on a gold price of $2,400/oz, a copper price of $4.00/lb, a silver price of $25.00/oz and an exchange rate of 1USD:1.33CAD.
- The Kemess Main open pit mineral resources (including the Nugget zone) are constrained by a pit shell and are reported based on a Net Smelter Return ("NSR") cut-off of $12.01/t (CA$15.97/t) that considers materials handling costs, metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges to determine economic viability. A dilution factor of 0% and a mining recovery of 100% is used.
- The Kemess South open pit mineral resources are constrained by a pit shell and are reported based on a NSR cut-off of $9.98/t (CA$13.27/t) that considers metallurgical recoveries, concentrate grades, transportation costs, and smelter treatment charges to determine economic viability. A dilution factor of 0% and a mining recovery of 100% is used.
- The Kemess Underground mineral resource is constrained by optimized stope shapes using commercially available software. Optimized stope shapes were included where the estimated average stope NSR exceeded a minimum stope cut-off value of $40.68/t (CA$54.10/t), representing the estimated breakeven value required to cover mining, processing, general and administrative, and sustaining capital costs. Economic screening was performed on stope shapes to ensure reasonable prospects for eventual economic extraction. Dilution was estimated using equivalent linear overbreak sloughing ("ELOS") for each slope type and ore-waste contacts, which vary between zero and 1.25 m. Mining recovery of 93% was applied to all stopes.
- The Kemess East underground mineral resource is constrained by optimized stope shapes using commercially available software. Optimized stope shapes were included where the estimated average stope NSR exceeded a minimum stope cut-off value of $40.68/t (CA$54.10/t), representing the estimated breakeven value required to cover mining, processing, G&A, and sustaining capital costs. Economic screening was performed on stope shapes to ensure reasonable prospects for eventual economic extraction. Dilution was estimated using ELOS for each slope type and ore-waste contacts, which vary between zero and 1.25 m. Mining recovery of 93% was applied to all stopes.
- The Kemess Main open pit shell was restricted to a minimum floor elevation of 1,355 metres above sea level ("masl") and the Kemess Underground optimized stope shapes were restricted to a maximum elevation of 1,355 masl, to represent the conceptual transition between open pit and underground mining zones for resource estimation purposes.
- A portion of the mineral resource estimate is included in the economic analysis for the PEA, which is limited to the Kemess Main open pit and Kemess Underground zones. This is a conservative subset that reflects mining, processing and economic assumptions. It is important to note that the PEA mining inventory is not a mineral reserve and does not demonstrate economic viability. The subset of the mineral resource used in the PEA was based on a gold price of $2,000/oz, a copper price of $3.75/lb, a silver price of $22.50/oz and an exchange rate of 1USD:1.33CAD.

14.14 GRADE SENSITIVITY ANALYSIS

Estimated mineral resources are sensitive to the selection of the reporting cut-off grade. In order to assess the sensitivity of the resource estimate to changes in cut-off grade, the block model quantities and grade estimates within the reporting solids used to constrain the mineral resources are presented in Table 14-36 to Table 14-39 at selected NSR values by resource category. These resource figures are inclusive of reserves and are compared to the base case MRE using a NSR cut-off value of CA$15.97/t for Kemess Main Open Pit, CA$54.10 for the KUG and Kemess East underground deposits, and CA$13.27/t for the Kemess South Open Pit deposit. The QP notes that the contained metals are relatively insensitive to NSR cut-off grades.

The reader is cautioned that the figures presented in these tables should not be misconstrued as a Mineral Resource Statement. The figures are presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-36: Kemess Main open pit Indicated and Inferred Mineral Resource sensitivity; base case scenario NSR cut-off CA$15.97/t

Kemess Main open pit – Indicated							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
15.97	**170,520**	**0.30**	**0.15**	**1.12**	**1,668**	**575**	**6,156**
18	161,633	0.31	0.16	1.15	1,624	562	5,991
20	154,079	0.32	0.16	1.18	1,585	548	5,829
22	147,940	0.33	0.16	1.19	1,550	534	5,682
24	141,551	0.33	0.17	1.22	1,509	520	5,530
26	133,957	0.34	0.17	1.24	1,455	503	5,348
28	125,178	0.35	0.17	1.27	1,389	480	5,119
30	115,552	0.35	0.18	1.30	1,310	454	4,844
32	103,901	0.36	0.18	1.34	1,209	420	4,482
Kemess Main open pit – Inferred							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
15.97	**237,068**	**0.30**	**0.13**	**1.06**	**2,300**	**682**	**8,110**
18	222,263	0.31	0.13	1.09	2,231	658	7,788
20	207,133	0.32	0.14	1.11	2,155	629	7,402
22	194,997	0.33	0.14	1.13	2,088	603	7,071
24	183,670	0.34	0.14	1.15	2,016	577	6,764
26	171,731	0.35	0.15	1.17	1,931	550	6,432
28	159,698	0.36	0.15	1.19	1,836	522	6,084
30	146,035	0.37	0.15	1.21	1,721	488	5,677
32	129,721	0.38	0.15	1.24	1,576	443	5,151

Table 14-37: KUG Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off CA$54.10/t

KUG – Indicated							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
54.1	**33,203**	**0.76**	**0.34**	**2.31**	**25,371**	**11**	**76,704**
59.1	32,920	0.77	0.34	2.31	25,247	11	76,192
64.1	31,527	0.78	0.34	2.33	24,626	11	73,582
69.1	28,852	0.81	0.35	2.38	23,280	10	68,640
74.1	26,602	0.83	0.36	2.42	22,085	9	64,300
79.1	24,922	0.85	0.36	2.43	21,104	9	60,653
84.1	22,965	0.86	0.37	2.46	19,859	8	56,427
89.1	20,939	0.88	0.37	2.47	18,403	8	51,779
94.1	19,296	0.89	0.38	2.48	17,202	7	47,783
99.1	17,378	0.91	0.39	2.49	15,870	7	43,298
104.1	14,995	0.94	0.39	2.51	14,027	6	37,670
109.1	13,228	0.96	0.40	2.52	12,654	5	33,340
114.1	10,859	0.97	0.41	2.58	10,586	4	28,041
119.1	8,916	1.00	0.42	2.61	8,920	4	23,286
124.1	7,234	1.03	0.43	2.63	7,443	3	19,007
129.1	5,304	1.08	0.44	2.65	5,728	2	14,042
KUG – Inferred							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
54.1	**20,113**	**0.74**	**0.33**	**2.22**	**14,958**	**7**	**44,586**
59.1	19,118	0.76	0.34	2.23	14,500	6	42,684
64.1	14,864	0.84	0.36	2.33	12,458	5	34,584
69.1	13,131	0.88	0.37	2.33	11,508	5	30,602
74.1	11,520	0.92	0.38	2.34	10,562	4	27,000
79.1	10,429	0.95	0.39	2.35	9,882	4	24,459
84.1	9,002	0.99	0.41	2.36	8,923	4	21,235
89.1	7,690	1.04	0.43	2.42	8,035	3	18,607
94.1	7,421	1.06	0.43	2.41	7,860	3	17,922
99.1	6,093	1.09	0.45	2.42	6,656	3	14,754
104.1	5,220	1.13	0.45	2.46	5,875	2	12,852
109.1	4,060	1.17	0.47	2.44	4,747	2	9,894
114.1	2,836	1.28	0.51	2.62	3,633	1	7,431
119.1	2,158	1.36	0.51	2.74	2,924	1	5,902
124.1	1,957	1.32	0.50	2.72	2,591	1	5,317
129.1	1,523	1.28	0.50	2.68	1,953	1	4,082

Table 14-38: Kemess East Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off CA$54.10/t

Kemess East – Indicated							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
54.1	**27,414**	**0.60**	**0.41**	**1.78**	**16,354**	**11**	**48,725**
59.1	27,292	0.60	0.41	1.78	16,321	11	48,481
64.1	26,290	0.61	0.41	1.78	15,941	11	46,842
69.1	23,716	0.63	0.42	1.79	14,883	10	42,418
74.1	20,594	0.65	0.43	1.82	13,373	9	37,509
79.1	18,746	0.67	0.43	1.83	12,538	8	34,327
84.1	16,482	0.69	0.44	1.82	11,378	7	30,078
89.1	12,445	0.71	0.45	1.87	8,870	6	23,284
94.1	9,638	0.73	0.45	1.88	7,034	4	18,164
99.1	5,045	0.79	0.46	1.96	4,005	2	9,897
104.1	2,332	0.86	0.46	1.96	1,994	1	4,571
109.1	935	0.91	0.48	2.10	850	0	1,963
114.1	611	0.88	0.49	2.14	541	0	1,309
119.1	283	0.91	0.53	2.44	257	0	690
124.1	128	0.94	0.55	2.41	121	0	309
129.1	112	0.96	0.55	2.42	107	0	272
Kemess East – Inferred							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
54.1	**42,329**	**0.57**	**0.42**	**1.91**	**24,022**	**18**	**80,918**
59.1	41,314	0.57	0.42	1.92	23,661	17	79,257
64.1	36,365	0.60	0.43	1.94	21,755	16	70,375
69.1	31,846	0.62	0.43	1.94	19,815	14	61,859
74.1	26,773	0.65	0.44	1.98	17,379	12	53,016
79.1	22,269	0.67	0.44	1.99	14,957	10	44,411
84.1	18,578	0.69	0.45	2.02	12,750	8	37,610
89.1	14,393	0.71	0.46	2.08	10,201	7	29,925
94.1	9,985	0.74	0.46	2.07	7,371	5	20,670
99.1	4,525	0.77	0.46	2.08	3,479	2	9,411
104.1	2,795	0.81	0.47	2.12	2,254	1	5,927
109.1	1,051	0.88	0.49	2.04	926	1	2,142
114.1	802	0.92	0.52	2.15	736	0	1,726
119.1	345	0.98	0.56	2.29	338	0	790
124.1	339	0.98	0.57	2.30	334	0	781
129.1	204	1.01	0.57	2.32	206	0	472

Table 14-39: Kemess South Open Pit Indicated and Inferred Mineral Resource sensitivity; base case NSR cut-off of CA$13.27/t

Kemess South Open Pit – Indicated							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
13.27	**13,204**	**0.37**	**0.13**	**0.68**	**158.3**	**37.5**	**289.2**
16	13,202	0.37	0.13	0.68	158.3	37.5	289.1
19	13,195	0.37	0.13	0.68	158.3	37.5	288.9
22	13,134	0.37	0.13	0.68	157.8	37.4	288.1
25	12,759	0.38	0.13	0.69	154.8	36.8	283.0
28	11,698	0.39	0.13	0.70	146.0	34.5	263.3
31	9,971	0.41	0.14	0.71	130.1	30.4	228.0
34	7,989	0.43	0.14	0.73	109.7	25.5	187.8
37	6,021	0.45	0.15	0.76	87.0	20.4	147.3
40	4,395	0.47	0.16	0.80	66.7	15.9	112.7
Kemess South Open Pit – Inferred							
NSR cut-off (CA$/t)	Tonnes (kt)	Grade			Material Content		
		Au (g/t)	Cu (%)	Ag (g/t)	Au (koz)	Cu (Mlb)	Ag (koz)
13.27	**198**	**0.34**	**0.08**	**0.42**	**2.1**	**0.4**	**2.7**
16	198	0.34	0.08	0.42	2.1	0.4	2.7
19	198	0.34	0.08	0.42	2.1	0.4	2.7
22	191	0.34	0.08	0.42	2.1	0.4	2.6
25	191	0.34	0.08	0.42	2.1	0.4	2.6
28	165	0.35	0.09	0.43	1.8	0.3	2.3
31	99	0.35	0.11	0.45	1.1	0.2	1.5
34	47	0.35	0.13	0.49	0.5	0.1	0.7
37	12	0.34	0.16	0.58	0.1	0.0	0.2
40	2	0.39	0.17	1.24	0.0	0.0	0.1

14.15 COMPARISON TO PREVIOUS MINERAL RESOURCE STATEMENT

The current MRE for the Kemess Main Zone project supersedes the previous estimate disclosed by Centerra in May 2025. The model comparisons are shown below in Table 14-40.

The increase in resource figures reflects a gold and copper price assumption of $2,400/oz and $4.00/lb, respectively, and incorporates additional ounces from the Nugget and Kemess South deposits resulting from technical work and recent drilling.

Table 14-40: Mineral Resources (December 31, 2025) comparison to April 15, 2025

Kemess Mineral Resources (1, 2)	December 31, 2025							April 15, 2025						
	Tonnes (kt)	Grade			Contained metal			Tonnes (kt)	Grade			Contained metal		
		Au (g/t)	Cu (%)	Ag (kt)	Au (koz)	Cu (%)	Ag (koz)		Au (g/t)	Cu (%)	Ag (kt)	Au (koz)	Cu (%)	Ag (koz)
Indicated Resources														
Open pit – Kemess Main	170,513	0.30	0.15	1.12	1,668	575	6,155	142,570	0.32	0.16	1.16	1,467	503	5,308
Open pit – Kemess South	13,204	0.37	0.13	0.68	158	38	289	-	-	-	-	-	-	-
Underground – KUG	33,223	0.82	0.36	2.48	877	265	2,652	25,347	0.91	0.39	2.60	745	217	2,122
Underground – Kemess East	27,491	0.64	0.44	1.91	565	268	1,684	25,074	0.66	0.45	1.94	531	251	1,564
Total Indicated	**244,431**	**0.41**	**0.21**	**1.37**	**3,269**	**1,146**	**10,780**	**192,990**	**0.44**	**0.23**	**1.45**	**2,742**	**971**	**8,994**
Inferred Resources														
Open pit – Kemess Main	237,050	0.30	0.13	1.06	2,299	682	8,108	124,428	0.31	0.14	1.06	1,232	395	4,228
Open pit – Kemess South	198	0.34	0.08	0.42	2	0.4	3	-	-	-	-	-	-	-
Underground – KUG	20,094	0.74	0.33	2.22	481	148	1,433	10,821	0.96	0.40	2.45	335	95	851
Underground – Kemess East	42,252	0.57	0.42	1.92	772	393	2,602	34,010	0.60	0.44	1.97	661	331	2,156
Total Inferred	**299,593**	**0.37**	**0.19**	**1.26**	**3,555**	**1,223**	**12,146**	**169,260**	**0.41**	**0.22**	**1.33**	**2,228**	**821**	**7,235**

NOTE: Totals may not sum due to rounding. (1) Refer to the Centerra Gold Updated Kemess Resource Summary table in section 14.13, including the respective footnotes and the “Additional Footnotes”. (2) The PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability

The previous Technical Report for Kemess was produced by Golder Associates Ltd in 2017, which focussed on the KUG and Kemess East deposits. Compared to the 2016 MRE, the current, updated estimate reflects changes resulting from additional drilling, updated geological interpretation, revised estimation parameters, the application of updated economic assumptions, and a change in mining methods. The Kemess Open Pit incorporates mineralized material previously considered part of a block cave concept for Kemess Underground, which has increased the size and quality of the resources considered for open pit mining. The tonnages and mineral inventory in KUG have decreased while grades have increased due to the Kemess Open Pit mining out a portion of the previously contemplated block cave, as well as the change of mining method to longhole open stoping. The tonnages in Kemess East also decreased while grades increased due to the lower dilution of longhole open stoping compared with the previous block cave mining method.

15 MINERAL RESERVE ESTIMATES

Mineral Reserves have not been declared as a result of the PEA.

16 MINING METHODS

16.1 INTRODUCTION

Following a decision to not develop an underground block cave mine, Centerra Gold/AuRico Metals has decided to investigate open pit mining and underground longhole stoping to extract the mineral resources of the Kemess Main Zone deposit.

This section provides information on geotechnical investigations, open pit mine design, underground mine design and the schedule to deliver run-of-mine (ROM) mineralized material to satisfy the proposed plant throughput rate.

The PEA Mine Plan is based on a subset of the Mineral Resource presented in Item 14. The plan focuses on mining the highest margin resources contained near surface in the Main Zone and the underlying higher-grade zone. The basis of the inventory used for the mine design is summarized in Table 16-1 and Table 16-2.

Table 16-1: Basis of mineral inventory for PEA Mine Plan

Item	Unit	PEA inventory basis
Gold price	US$/oz	$2,000.00
Copper price	US$/lb	$4.00
Silver price	US$/oz	$0.00
Exchange rate	CA$:US$	$1.33
Mineral Resource classifications		Indicated and Inferred
Processing cost	CA$/t milled	$8.10
Processing method		exclude leach circuit
Open pit cut-off NSR (for pit design)	CA$/ore tonne mined	$13.47
Base open pit mining cost	CA$/t mined	$3.50
Underground cut-off NSR	CA$/t mined	$56.10
Underground Cut-off NSR – development	CA$/t mined	$14.60

Table 16-2: PEA Mine Plan mineral inventory

Mining technique	Mineral Resource classification	Tonnes (Mt)	Grade			Contained metal		
			Au g/t	Ag g/t	Cu %	Au koz	Ag koz	Cu Mlb
Main Zone Open Pit	Indicated	130	0.32	1.14	0.16	1,333	4,769	454
	Inferred	94	0.30	1.02	0.14	905	3,079	296
Underground	Indicated	22	0.93	2.58	0.39	644	1,785	186
	Inferred	9	0.98	2.36	0.40	293	696	80
Kemess Mine Plan		255	0.39	1.26	0.18	3,176	10,329	1,017

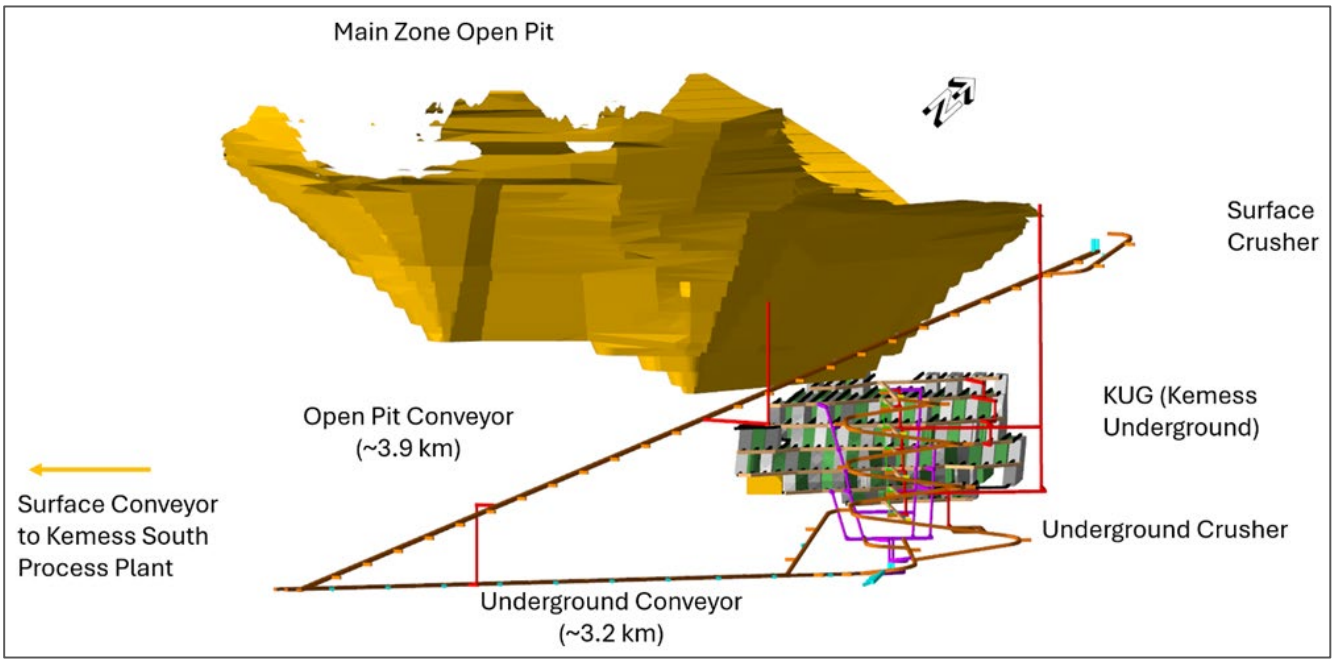
16.1.1 Combined Open Pit and Underground Mining Concept

The PEA Mine Plan is comprised of a combined open-pit and underground mining strategy operating simultaneously. It is assumed that the mine will operate 365 days a year, with two 12-hour shifts per day. The mining operations are focused in the Kemess North valley, located 5 km directly north of existing camp facilities. Initial access to Kemess North is planned from the west, using upgraded existing road infrastructure.

Mining is planned to begin with the Main Zone Open Pit. After mill commissioning, the decline construction to KUG will commence. Conveyor tunnels will be used to transport ROM mineralized material to the existing Kemess South Processing Plant. This concept is illustrated in the figure below. The PEA Mine Plan assumes that the crown pillar between pit bottom and the uppermost underground stopes will be 100% recoverable near the end of mining.

The operations strategy assumes owner-operated mining operations. Initital underground mine development and open pit pioneering works will be carried out by contractor.

Figure 16-1: Isometric view of the integrated open pit and underground concept for the Kemess PEA



16.2 GEOTECHNICAL INVESTIGATIONS

Historical geotechnical information and recent geotechnical investigations have informed pit slope design criteria and design of underground excavations. Rock property and testing data have been collected for a number of years to inform previous engineering studies. Test results and engineering evaluations pertinent to open pit mine design and underground mine design will be discussed separately in this report.

The project area has been the subject of multiple technical studies and development initiatives for both open pit and underground mining methods. As a result, a significant amount of technical information is available.

The geomechanical drillhole dataset has been compiled from five distinct site investigation programs:

- 2003 – KP: 7 oriented core drillholes with detailed RMR89 logging, targeted at the Main open pit.
- 2010 – AMC: 29 exploration drillholes with simplified Q' logging, 26 of which are oriented core drillholes, targeted at the underground deposit.
- 2011 – SRK: 7 oriented core drillholes with detailed RMR90 and IRMR (after Laubscher & Jakubec, 2000) and Q' logging, 3 televiewer surveys (for decline only), and hydrogeological data collection by Lorax, targeted at the underground deposit and decline.
- 2020 – SRK: 5 oriented core drillholes with detailed RMR76, RMR89 and Q' logging (individual logged parameters not available), televiewer surveys and packer testing targeting the portal and decline.
- 2024 – KP: 6 oriented core drillholes with detailed RMR89 and Q' logging, and 5 televiewer surveys, targeted at the Main and Nugget open pits.

16.2.1 Open Pit Geotechnical Investigation

Geohazards Assessment

In early 2025, KP delivered a letter report entitled "*Kemess Restart Project – Preliminary Geohazards Assessment for Proposed Pit Area*, VA25-00024, February 27, 2025." The assessment reviewed the geology, rock types, properties and condition, topography, and geomorphic features. Terrain hazards were identified and an assessment made of rockfall hazard zones.

The terrain analysis indicated extensive rockfall hazards at the site, primarily occurring on the cirque escarpments. Recent and relict rockslides were mapped on the escarpments. It is interpreted that rockslides transition to rockfalls after intersecting the talus slope and the largest rockfalls occurring on the talus slope initiated as rockslides.

Freeze-thaw is expected to be a key process contributing to rock falls on the escarpments, in particular during the Spring and late Summer/early Fall. It is possible rockfall susceptibility was exacerbated by permafrost degradation and that the process may still be having an adverse influence.

An empirical rockfall run-out assessment was undertaken for the largest source volume estimate of 6,400 m^3 using the angle of reach and the shadow angle methods. The rockfall run-out assessment shows the angle of reach approach yields a larger maximum run-out than the shadow angle approach

for cases where the steep rock slope source zone has a height in excess of approximately 50 m. The approach used for assessing the hazard zones is possibly conservative as it is assumed there is potential for a rock fall to initiate at the crest of the escarpments along their full lengths. This assumption is not, however, considered to be overly-conservative in the case of the Kemess Project site as there is an almost continuous talus slope beneath the cirque escarpments and the largest rockfalls are observed to have initiated at the crests of the escarpments.

The distance the rockfall hazard zone extends beyond the toe of the talus slope (the down slope extent of the 'rockfall shadow') is generally in the range of 10 m to 40 m. The run-out assessment shows there is potential for enhanced run-out of rockfalls that intersect a drainage line.

The escarpment at the Main Pit is located south of the footprint with the south wall of the proposed Main Pit extending across the talus slope. The escarpment at the Main Pit will remain in place throughout pit operations and the risk to workers will increase as the south wall extends closer to the escarpment. An additional landslide hazard would be introduced as the South Wall of the Main Pit encroaches into the talus slope. It is likely that development of the Main Pit footprint, as proposed, would necessitate removal of talus material in addition to stabilization of the escarpment, prior to development. Alternatively, the rockfall and boulder fall risk at the Main Pit could be mitigated by designing the South Wall with a flatter overall slope so that the crest is located behind the crest of the escarpment.

Careful monitoring of the rockfall hazards will be needed throughout the development and operation of the Project with periodic inspections being undertaken by a registered specialist.

Open Pit Geotechnical Site Investigation

In 2024, KP was retained by AuRico to complete an open pit geotechnical site investigation program for the Project. The program advanced a drilling investigation targeting the proposed Nugget Pit and Main Pit at the Kemess Main Zone deposit to support a PEA in early 2025. Only the results of the Main Pit investigation are reported herein. KP reported its findings in the factual data report "*2024 Open Pit Geotechnical Site Investigation*, VA101-4/27-1, December 11, 2024".

The objectives of the program included:

- Collection of rock mass data and geotechnical characterization for the subsurface conditions of the anticipated final pit walls
- Collection of oriented discontinuity data to support rock mass structural characterization for open pit slope design
- Collection of hydraulic conductivity data to assess dewatering prospects and requirements for the open pit

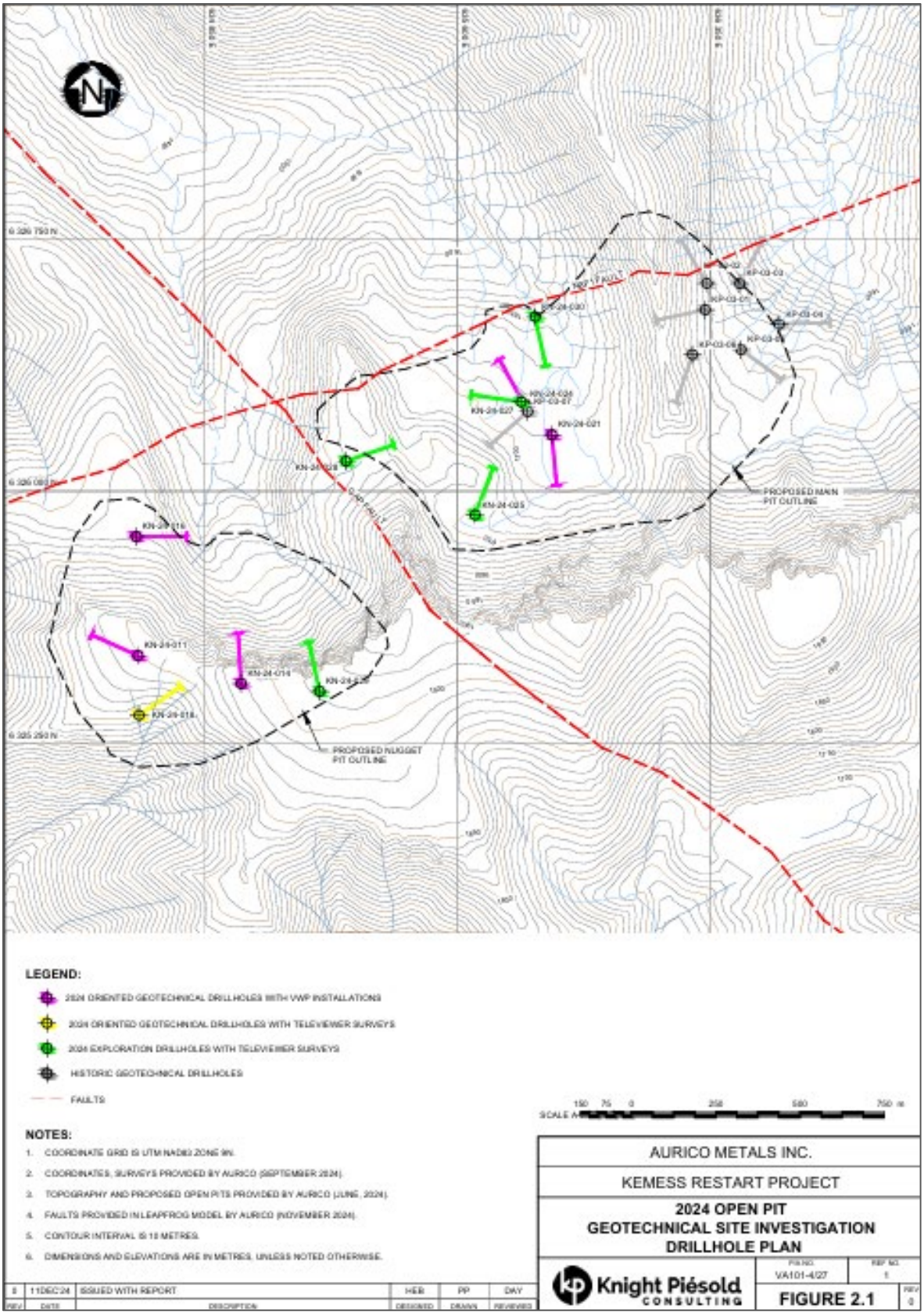
- Installation of vibrating wire piezometers (VWPs) to characterize and monitor groundwater levels and subsurface pore water pressures within the planned project area.

The site investigation program was completed from July to September 2024 using three alternating diamond drill rigs. The scope included detailed geomechanical logging with core orientation, rock core sample collection, field and laboratory strength testing, hydraulic conductivity testing, and instrumentation installation. The program objectives were achieved through completion of the following scope:

- Oriented core drilling, geomechanical logging, core orientation measurement, and sampling of six geotechnical drillholes. Point Load Testing (PLT) was completed on samples from all drillholes with 16 representative core samples collected for laboratory strength testing.
- Constant head testing using a hydraulic packer system and open hole testing in zones where a packer could not be seated. A total of 49 successful hydraulic conductivity tests were completed to support the open pit hydrogeologic characterisation.
- Installation of nested VWPs in five geotechnical drillholes. A total of 20 sensors were installed with 4 sensors nested within each VWP installation site. 17 VWPs remain functional at the end of the field program.
- Televiewer surveys were completed in five exploration drillholes to collect additional rock mass structural data. A televiewer survey was completed in one oriented geotechnical drillhole for calibration between televiewer surveys and oriented core measurement.

Figure 16-2 (Figure 2.1 of the KP report) identifies the location of the geotechnical drill holes and VWP installations.

Figure 16-2: Geotechnical drillhole locations



The 2003 UCS testing results can be found in the KP 2004 feasibility pit slope design report (KP, 2004) and the 2024 testing results are presented in the KP 2024 SI factual data report (KP, 2024).

A total of 30 Uniaxial Compressive Strength (UCS) tests were completed on samples from the Kemess North Main Pit area in 2003. The selected samples represent a variety of rock and alteration types encountered in the area. Mirarco Sudbury conducted all 2003 UCS tests for samples collected from the proposed Main Pit area. A summary of the laboratory UCS testing results is presented in Table 16-3.

Table 16-3: Laboratory UCS testing summary

Area	Unit	No. of samples	Mean strength (MPa)	Median strength (MPa)	Standard deviation (MPa)
Main Pit	Takla Group	19	63	62	32
	Hazelton group	5	131	126	71
	Post Mineralization	2	313	313	19
	Black Lake Intrusives	4	68	60	34

Mirarco Sudbury conducted seven laboratory direct shear tests on rocks from the Kemess North deposit area during the KP 2003 program. A summary of the laboratory direct shear test results is presented in Table 16-4.

Table 16-4: Laboratory shear testing summary

Source	Unit	Peak friction angle (°)	Residual friction angle (°)	No. of samples
2003 Direct Shear Tests	Takla Group	51	38	2
	Hazelton Group	45	34	1
2003 Saw Cut Direct Shear Tests	Takla Group	37	-	1
	Hazelton Group	40	-	1

KP recommends that the installed VWP sensors should continue to be monitored, and the data be reviewed to support hydrogeological characterization and long-term pit dewatering plan. A complete record of all VWP data should be established and maintained.

The geotechnical data collected from the 2024 SI program is sufficient for a PEA level open pit slope design study and addresses the main gaps from previous geotechnical site investigations primarily in the western part of the proposed Main Pit area. Additional geotechnical drilling, surface mapping, geotechnical model refinement, and hydrogeological model development will be required should the Project advance to prefeasibility study and/or feasibility study.

Open Pit Slope Design

Following completion of the terrain geohazard assessment and geotechnical assessment, KP combined that knowledge with updated geology and fault structure models for geotechnical characterization. These input parameters were used to create a preliminary geotechnical model for slope stability analysis and pit slope design. The maximum depth of the proposed ultimate pit is in the order of 400 m.

Geotechnical Characterization

The Kemess property is located at the northeast margin of the Stikine arc terrane, which spans much of western British Columbia, and at the southern end of the Toodoggone gold-copper mineral district, a 100 km long and 30 km wide north-northwest trending belt of mineral deposits and prospects hosted in Mesozoic era volcanic rocks and coeval intrusive rocks. The main geological units encountered include the Takla and Hazelton volcanic rocks, as well as the Black Lake and Late Triassic intrusive bodies. A rock mass model was developed for geotechnical characterization. The model includes the following geotechnical units. The overburden has limited exposure and was not the focus of the study.

- Takla Group – comprised of massive coarse grained basaltic porphyries and is the dominant unit in the deposit.
- Hazelton Group – comprised of reddish pink sub-aerial andesitic and dacite tuffs and lava flows.
- Black Lake Intrusives – comprised of quartz monzonite and quartz diorite.
- Post Mineralization Intrusives – comprised of quartz monzonite, quartz monzodiorite, quartz feldspar porphyry, and mafic volcanic intrusions.
- Broken Zone – comprised of a highly fractured poor rock mass quality zone near surface in the Main zone deposit.

Geotechnical rock mass characteristics for five geotechnical units are summarized in Table 16-5.

Table 16-5: Summary of rock mass characteristics

Item		Nugget Pit and Main Pit Combined				
		Broken Zone	Takla Group	Hazelton Group	Post Mineralization Intrusives	Black Lake Intrusives
Field Estimated UCS ALL (MPa)	Number of Samples	363	2006	239	45	117
	Mean	17	63	87	79	74
	Median	10	60	100	90	80
	30th Percentile	3	45	80	60	70
	Standard Deviation	22	28	40	24	20
Laboratory Testing UCS ALL (MPa)	Number of Samples	0	23	6	2	5
	Mean	-	66	119	313	60
	Median	-	62	126	313	52
	30th Percentile	-	50	90	307	39
	Standard Deviation	-	35	70	19	35
PLT UCS ALL (MPa)	Number of Samples	7	624	124	32	81
	Mean	60	80	141	127	115
	Median	61	64	110	119	80
	30th Percentile	61	39	50	71	35
	Standard Deviation	7	64	125	84	120
Intact Rock Description		MEDIUM STRONG	STRONG	VERY STRONG	VERY STRONG	STRONG
RQD ALL	Number of Runs	382	24114	3158	1011	5746
	Mean	14	64	52	56	66
	Median	0	73	53	57	71
	30th Percentile	0	52	35	41	55
	Standard Deviation	25.1	32	29	26	28
RMR by Run ALL (Logged Geotechnical Drillholes)	Number of Samples	380	2012	239	45	117
	Mean	31	62	54	54	64
	Median	27	64	55	54	62
	30th Percentile	26	54	52	52	58
	Standard Deviation	10	16	10	7	13
RMR Rating Description		POOR	GOOD	FAIR	FAIR	GOOD

Intact rock strength of the volcanic rock package ranges from Strong to Very Strong, except in the Medium Strong Broken Zone unit. Overall rock mass ranges from Fair to Good quality and is largely competent outside of the Broken Zone. Localized east-west and northwest trending faults and lithology contacts were identified but will require further verification. Three rock mass structural domains were delineated for the geotechnical study.

Hydrogeology

The groundwater flow regime in the deposit area is characterized by an overall northward flow originating from the ridges along the southern edge of the deposit.

Unconfined groundwater flow in the Broken Zone and relatively impermeable rock at depth generally characterize the hydrogeology of the open pits. The complex pattern of faulting, intrusion, and alteration in the Kemess North deposit has resulted in considerable spatial variability in rock mass permeability.

KP conducted downhole packer permeability tests during the 2003 and 2024 SI programs. Three hydrogeological domains, including the Broken Zone, Takla Group, and Hazelton Group, were defined for this scoping-level study, based on all available testing data. The range and geometric means of the measured hydraulic conductivities for each hydrogeological unit are summarized in Table 16-6.

Table 16-6: Summary of rock mass permeabilities

Hydrogeological unit	No. of tests	Arithmetic mean (m/s)	Geometric mean (m/s)	Minimum (m/s)	Maximum (m/s)
Broken Zone	10	1E-06	8E-07	9E-08	5E-06
Takla Group	70	3E-07	2E-08	3E-11	6E-06
Hazelton Group	10	8E-07	1E-07	1E-09	5E-06

The hydraulic conductivities of the Broken Zone are relatively lower than those of highly fractured rock mass, indicating a moderate draining pace in this unit. However, preferential flow paths in higher permeability rock are anticipated in the Broken Zone based on loss of water circulation observed during drilling at various locations and depths during the 2003 and 2024 geotechnical SI programs.

Open pit development will have a significant impact on the local hydrogeologic regime, as the open pit will become a groundwater discharge zone. Piezometers suggest that the existing groundwater table varies between 2 mbgs and 60 mbgs. Progressive development of the pit will result in a gradual lowering of the groundwater table in the vicinity of the pit excavation. The elevated groundwater table with respect to the pit floor influences the mine development in that groundwater inflows need to be pumped out of the pit, groundwater depressurization measures are required to enhance pit slope stability, and production blasting must consider the effects of wet blastholes. Progressive development of the underground mining operation will also likely contribute to the gradual lowering of the groundwater table.

Whether the known shear zones or faults will act as conduits or barriers to groundwater flow will depend on their location relative to the open pit excavation and on the permeability contrast between the fault zones and the surrounding rock mass. Excavation of the open pit in zones of higher fracture density and correspondingly higher permeability will result in natural drainage and pressure relief by gravity.

Slope Stability Analyses

Design sectors were delineated for the proposed Main Pit based on geological and structural features, rock mass characteristics, and the orientations of the proposed ultimate pit walls. The design methods used to determine appropriate pit slope angles included a kinematic stability assessment and an evaluation of overall rock mass stability. Pit slope geometries for each design sector were determined

based on the minimum acceptable factors of safety and/or probability of failure criteria established for each design method.

Stereographic analyses were performed to determine the potential kinematic failure modes and their probability of occurrence in each design sector. Given a generally competent rock mass, steep bench face and inter-ramp slope angles can be kinematically achieved where adverse structural features are absent. However, slightly flatter slopes are recommended for the Main Pit West sector due to the potential adverse features of rock mass structures.

The limit equilibrium analyses were conducted for the highest pit walls with representative geological profiles to calculate global factors of safety against multiple-bench, large-scale slope failures through the rock mass. These analyses indicate that the kinematically determined slope angles are suitable, and that rock mass strength is not a controlling factor for pit wall stability with an exception for the Broken Zone.

Pit Slope Design Criteria

Slope design criteria for the proposed open pits were developed from site-specific geotechnical data, geological models, and corresponding slope stability analyses. Recommended pit slope configurations, including the bench face angles (BFAs), inter-ramp angles (IRAs), and overall slope angles (OSAs) for the proposed open pits are summarized in Table 16-7. Overburden slopes and natural slopes above the pit rim are not included in the OSA estimates.

Table 16-7: Recommended pit slope configurations

Pit design sector	Nominal wall dip direction (°)	Overall slope height (m)	Geotechnical domain	Bench height (m)	Bench width (m)	BFA (°)	IRA (°)	No. of step-outs or ramps	OSA (°)
All	-	-	Overburden	15	8	40	30	-	-
Main Pit West	030	300	Broken Zone	15	8	60	42	2	41
			Takla	30	16	65	45		
Main Pit East	280	400	Takla	30	15	70	49	2	44
Main Pit North	150	400	Takla/Hazelton	30	14	70	50	2	45

Notes:

- Most representative slope height in each design sector is shown.
- A maximum inter-ramp slope height of 150 m is recommended for bedrock slopes.
- OSAs are variable based on slope height, applied IRA, and number of ramps or step-outs.
- A 30 m wide ramp and/or step-outs are assumed for OSA estimation.
- Overburden slopes and natural slopes above the pit rim are not included in OSA estimation.

The structural fabric is moderate to steeply dipping, and the potential for wedge failure has been identified in most of the design sectors. A BFA ranging from 65° to 70° can be achieved in all the sectors of the proposed pit.

A 15 m single bench height is assumed for open pit development. Double-benching configurations (2 x 15 m high) are recommended for rock slope development outside of the Broken Zone and overburden. The resulting IRA for single benches in the Broken Zone is 42°, while steeper IRAs ranging from 44° to 50° are recommended for the double-benching pit walls formed by competent rocks.

It is recommended that the maximum inter-ramp slope height not exceed 150 m for potential rockfall hazard management. A geotechnical step out (wider bench) or haul ramp, assuming 30 m width, should be placed on inter-ramp slopes exceeding 150 m in height to effectively flatten the OSAs and to provide additional capacity for rockfall containment and debris cleanout access. The resulting OSAs are typically a few degrees flatter than the applied IRAs and may vary with the slope height, step-out/haul ramp locations, and overburden thickness in the pit walls.

The implementation of the slope design in accordance with the mine plan not only includes specifications for slope geometry but also requires low-damage, well-controlled blasting and excavation practices, effective pit dewatering and slope depressurization, and systematic slope monitoring throughout pit operations.

Geohazards Considerations

The Main Pit footprint, as proposed, would necessitate removal or flattening of the talus slope and stabilization of the escarpment to mitigate the debris slide, rockfall and boulder fall risks to workers. The following preliminary design options have been considered to mitigate these terrain hazards:

- Designing the affected pit slope to be behind the escarpment which means removal of all talus material and mining out the escarpment rock face,
- Stabilization of the escarpment and removal of the talus material prior to pit development, or
- Partial excavation and re-profiling of the talus slope. This option would have the largest requirement for management of landslide hazards throughout the life of the Project.

The current open pit design considers removal of a large portion of the talus slope and excavation of a section of the escarpment rock face. In addition, geotechnical berms for control of rockfall and snow removal have been incorporated into the design.

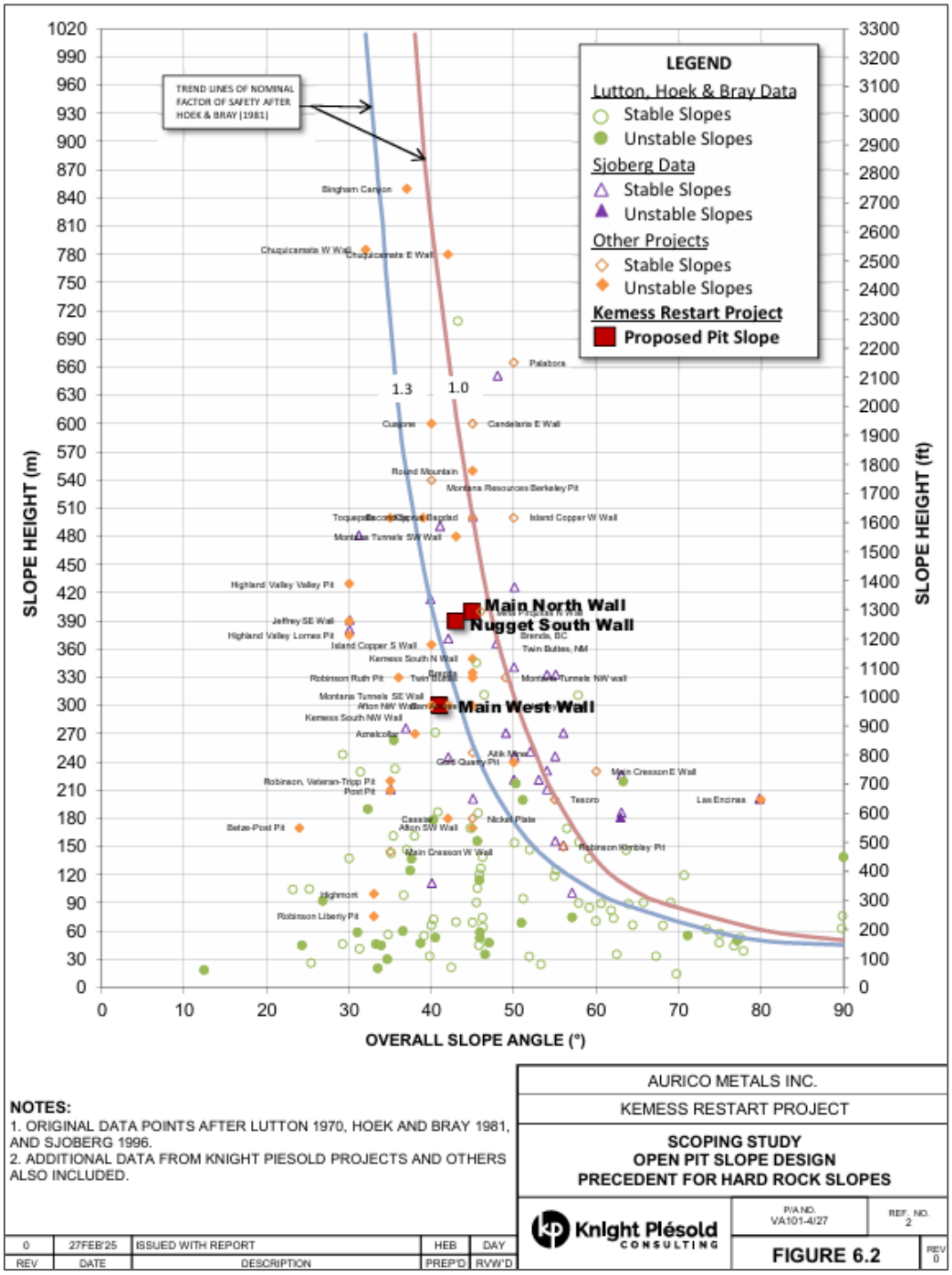
Precedent Practice

Pit slope stability depends on a variety of site-specific factors (such as geologic structure, alteration processes, rock strengths, groundwater conditions, discontinuity strength and orientation, pit geometry, blasting practices, stress conditions, climatic conditions and time), which make it difficult to provide direct comparisons with other operating mines. However, it is still useful to review the successes and problems

encountered at other operations to recognize opportunities and potential constraints for the proposed open pit development.

The proposed slope angles for the Main Pit are generally comparable to the slope angles achieved in other deep pits. A summary plot of pit depth vs. slope angles achieved in various operations is illustrated on Figure 16-3 (Figure 6.2 of the KP report).

Figure 16-3: Precedent for hard rock slopes



Pit Slope Design Summary

The geotechnical data collected from various SI programs is considered sufficient for a scoping level open pit slope design for the Project. The corresponding stability analyses suggest the recommended pit slope angles are reasonable and appropriate. However, there are data gaps and uncertainties relating to large scale structural features (faults), rock mass structure, zones of weak rock mass, and consequences of excavating the toe of the existing talus slopes above the proposed ultimate pit walls.

Further geotechnical data collection, rock mass characterization, and slope stability assessment are required should the Project advance to prefeasibility study or feasibility study. See Item 26 for recommended work in support of a PFS.

16.2.2 Geotechnical and Hydrogeological Considerations for Underground Mining

Numerous studies have been completed for potential open pit and underground mining of the Kemess/Kemess North deposits since the early 2000s. The following summaries from the geotechnical reviews are provided that support the mining assumptions considered in the underground mine plan.

Previous Studies

Geotechnical Characterization (SRK, 2012)

The SRK Consulting report (2012) is considered the most comprehensive source of geotechnical information for the project. After review, the QP considers the report acceptable for use in the current PEA. Most subsequent studies utilize the geotechnical information and parameters from this report as inputs.

Geotechnical Dataset

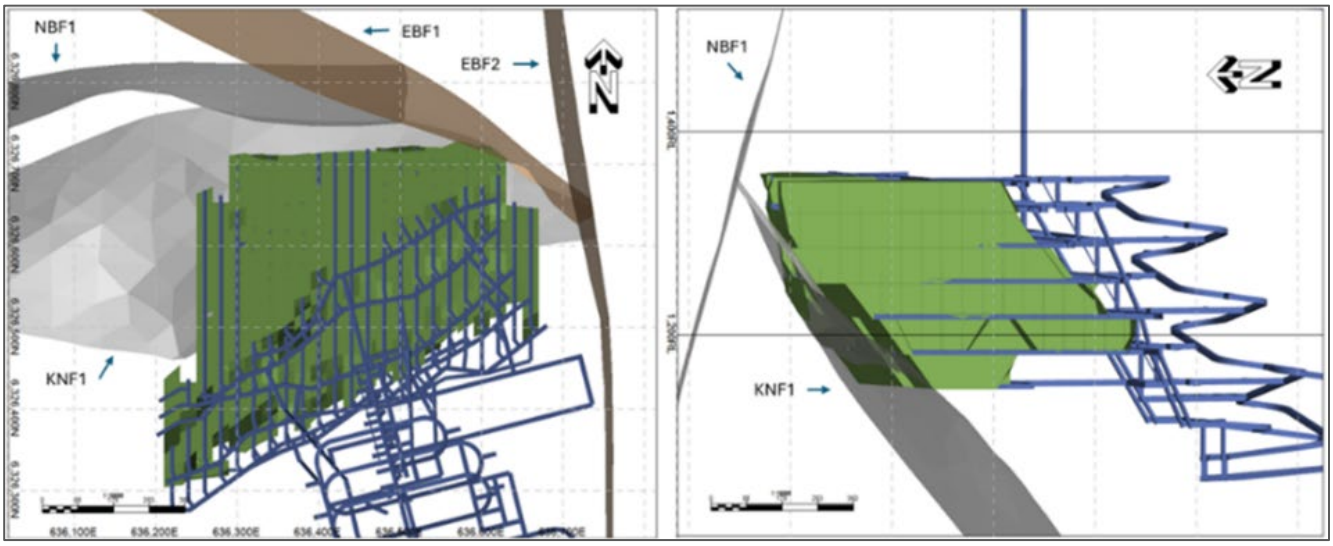
Geotechnical data has been collected on most diamond drill holes within the planned mine footprint, forming the basis of the geotechnical dataset. Some areas in the planned mine design have poor geotechnical data available and should be targeted in future diamond drilling campaigns. The amount of data within the mining area is sufficient basis for this preliminary economic assessment.

Major Structures

Three major faults are identified within the proposed underground mine area. These faults are show in Figure 16-4. The Kemess North Fault (KNF1) is the most prominent structure. This thrust fault intersects at the hanging wall of the stoping area. 9% of stopes are in the footwall zone of the fault, and mining is proposed to retreat out of this area. KNF1 serves as a lithological contact between the competent Hazelton rock unit and the mineralized Black Lake unit. The damage zone associated with KNF1 is 60 m to 80 m wide and consists of fault breccia and gouge. The North Boundary Fault (NBF1) truncates the

Kemess North Fault in the northeast. The third structure is the East Boundary Fault (EBF1) which truncates KNF1 and NBF1 beyond the easternmost extents of the mineralisation.

Figure 16-4: Major structures affecting underground design



SRK (2012) identified a further zone of weakness (through core fracture and recovery logging) that is parallel to the KNF1 fault and in the top contact of the black lake mineralization. The contacts in this zone may be weak due to alteration or shearing.

Rock Properties

Samples were collected during the 2011 SRK logging program and selected to be representative from each lithology and alteration type encountered. The results were also referenced on a contextual basis against tested samples from previous geotechnical investigations, namely the 2003 KP and 2010 AMC testing programs.

Uniaxial compressive strength testing was used to assign strengths to each geotechnical domain. No comprehensive in-situ stress testing was performed, and it is identified that further stress measurements will be required at the next study phase.

Ground Support

For most of the infrastructure, stability assessments and support requirements are based on empirical estimates (Grimstad & Barton, 1993, Laubscher, 1990) modified with experience in other operations with similar conditions. This is considered appropriate for a PEA. For more detailed studies (prefeasibility or feasibility level studies), detailed kinematic and numerical analysis is required.

Geotechnical Database

The Kemess Project has diamond drilling data collected over several decades to support both resource definition and the design of open pit and underground mines. Following a review of the available information, it was determined that not all datasets meet the standards required for the geotechnical characterization needed in the current design.

For this preliminary economic assessment, the datasets from the 2001, 2002, 2010, and 2011 drilling campaigns were selected due to the quality of the recorded logging parameters. These records adequately cover the footprint of the mineralized body.

Rock Mass Characterization

The data processing workflow involved compositing the drillhole logs into 1 m intervals and segregating them according to geological and infrastructure models provided by Centerra. The domain structure included:

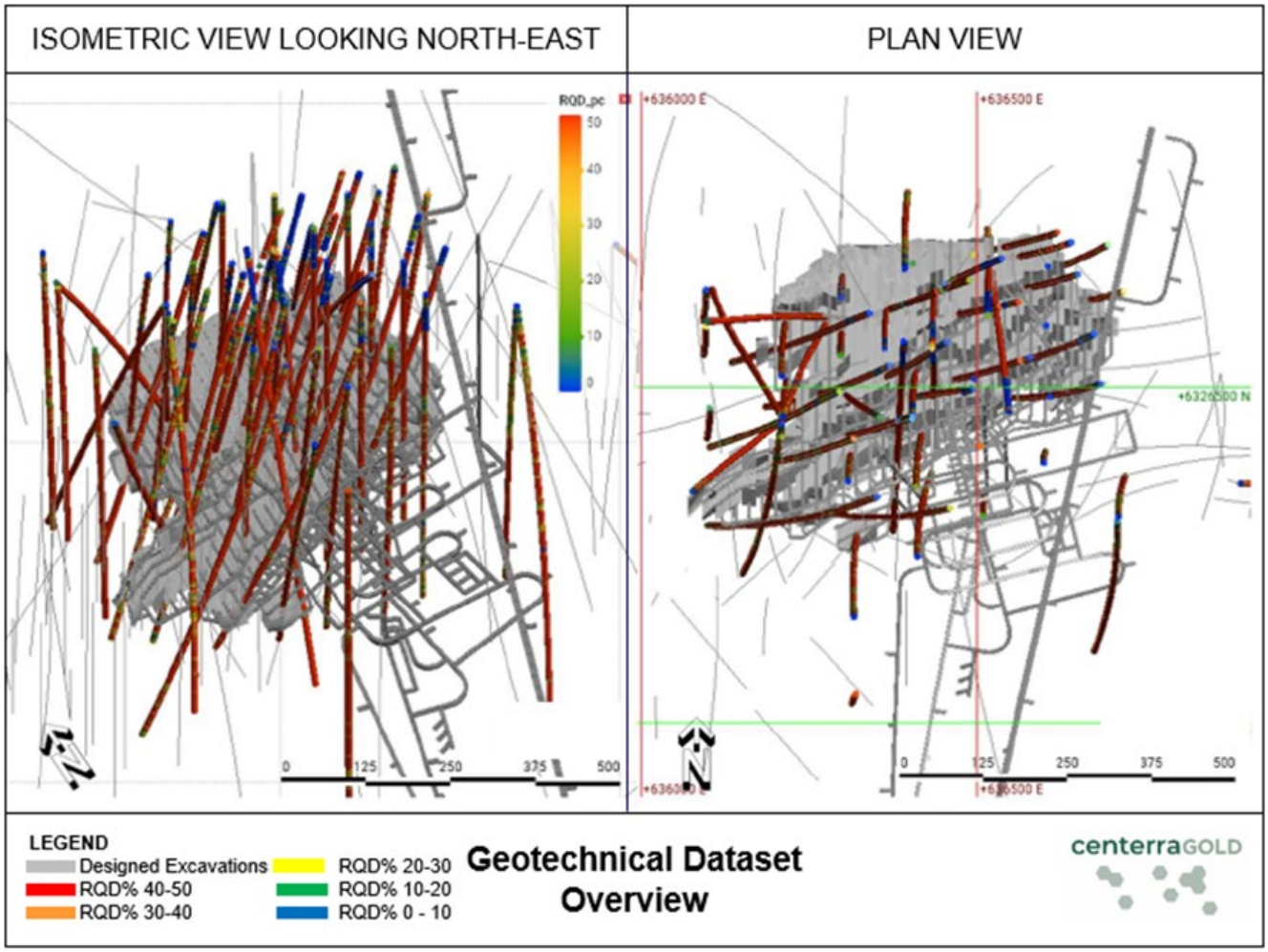
- Geological Domains: Defined by lithology (Black Lake, Takla, Hazelton, etc.) and alteration zones (Potassic, Phyllic, Propylitic)
- Structural Domains: A dedicated domain was established for the Kemess North Fault (KNF) by applying a 10 m buffer around the KNF1 modelled fault surface in the footwall.

Rock Mass Variability

The geotechnical model demonstrated that the rock mass exhibits spatial variability that does not correlate directly with lithology or alteration. Zones of competent rock were identified alongside areas of very poor quality.

A distinct Poor Ground Zone was identified in the central portion of the footwall of the proposed preliminary mine plan. RQD analysis allowed this zone to be modelled using isometric surfaces representing RQD <25% and RQD between 25% and 50%. These locations are illustrated in Figure 16-5. No obvious structural features or lithological contacts were identified to explain this condition, suggesting a potential relationship with the intersection of major fault structures.

Figure 16-5: RQD% iso-surfaces in the Mine Plan



Stope Sizes

The assessment of the maximum permissible dimensions for unsupported stopes was completed using the Modified Mathews–Potvin Stability Graph Method. This empirical method calculates the Modified Stability Number (N') based on the Q' index (derived from RMR). The analysis focused on determining the allowable Hydraulic Radius (HR) for a fixed stope height of 35 m (level spacing), which then allowed estimation of the maximum sustainable strike length for the sidewalls and the crown.

Mathews et al. (1981)

$$N' = Q' x\ A\ x\ B\ x\ C$$

Where:

- *A = Rock stress factor accounting for in-situ and mining induced stresses,*
- *B = Joint orientation factor to adjust for the weakness caused by the orientation of the dominant joint set, and*
- *C = Surface orientation factor to identify the directional influence of the exposed surface and gravitational effects.*

The stability assessment was carried out for three mining depths (450 m, 550 m, and 670 m) to capture variations in the stress environment:

- Expected conditions (~85% of stopes):
  - In the Black Lake domain, the current design dimensions (20 m wide × 20 m long) are considered stable, allowing strike spans of up to 29 m at 450 m depth and 23 m at 670 m depth.
  - The Takla–Phyllic domain showed the lowest stable spans due to its lower intact strength, limiting the strike length to approximately 15–18 m.
- Lower bound conditions (~8% of stopes):
  - In these areas, maximum stable strike length is reduced to avoid increased overbreak/dilution.
- Poor ground conditions (~7% of stopes):
  - In these areas, the maximum stable strike length is significantly reduced, ranging from approximately 8 m to 13 m.

The standard stope size used for this Preliminary Economic Assessment is 20m x 20m x 35m. Historic RMR89 data was converted to the Barton Q system prior to completing a stope stability assessment on the proposed geometry. The assessment followed the Modified Mathews–Potvin Stability Graph Method with a fixed 35m stope height. From this empirical assessment, it appears that this stope size is achievable for 85% of the planned stopes.

Adjustments will be required to the design and extraction sequence around the known poor ground and further work is required to better define this zone and any other such zones identified. Figure 16-6 plots the adjusted hydraulic ratios (HR) required by ground conditions area to achieve stability. -6

Figure 16-6: KUG stope designs and depths plotted on the Modified Stability Graph

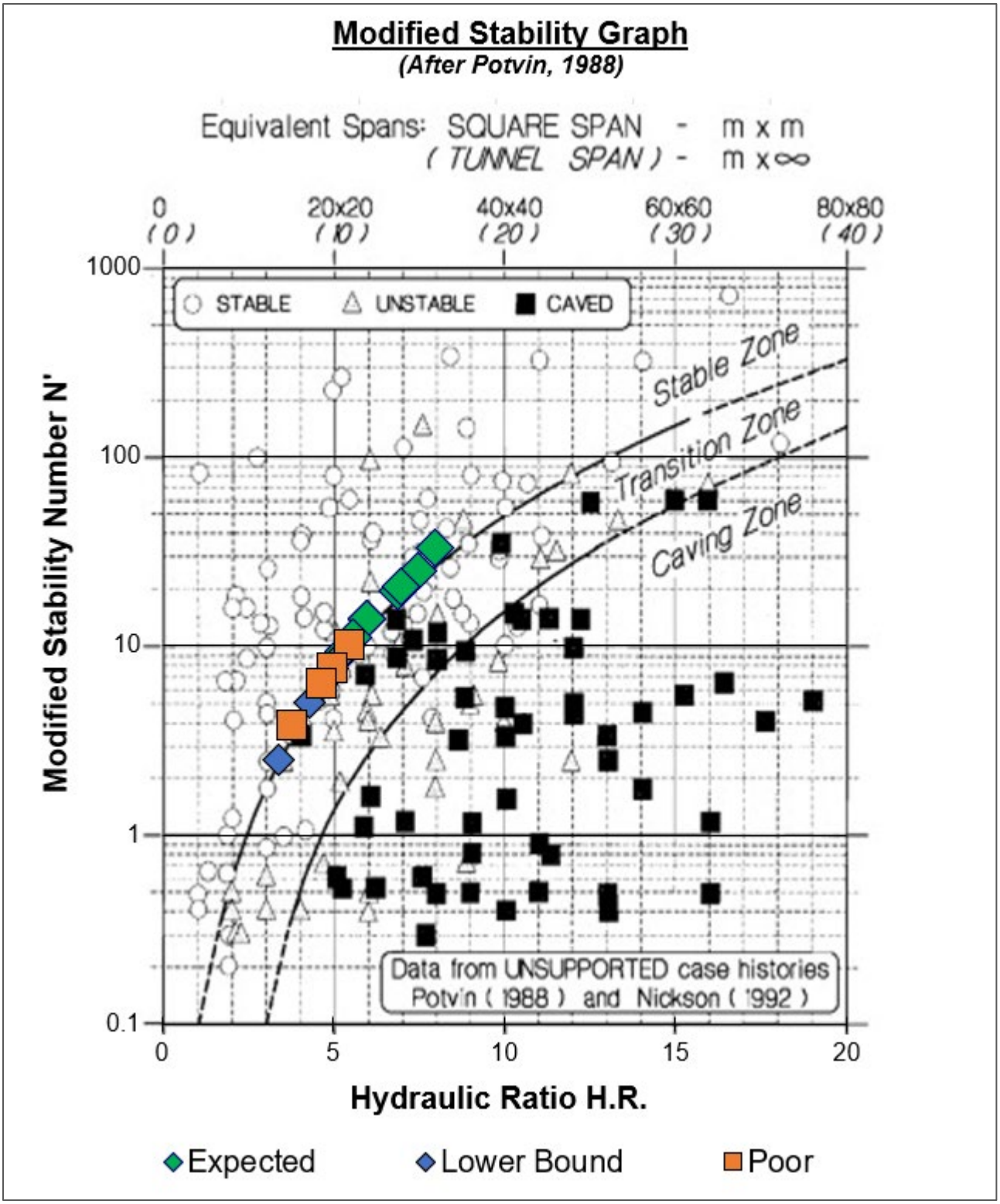


Table 16-8 summarises the various strike lengths for the three major rock units at increasing depth required for stability.

Table 16-8: Calculated maximum strike sidewall lengths in all ground conditions at various mining depths

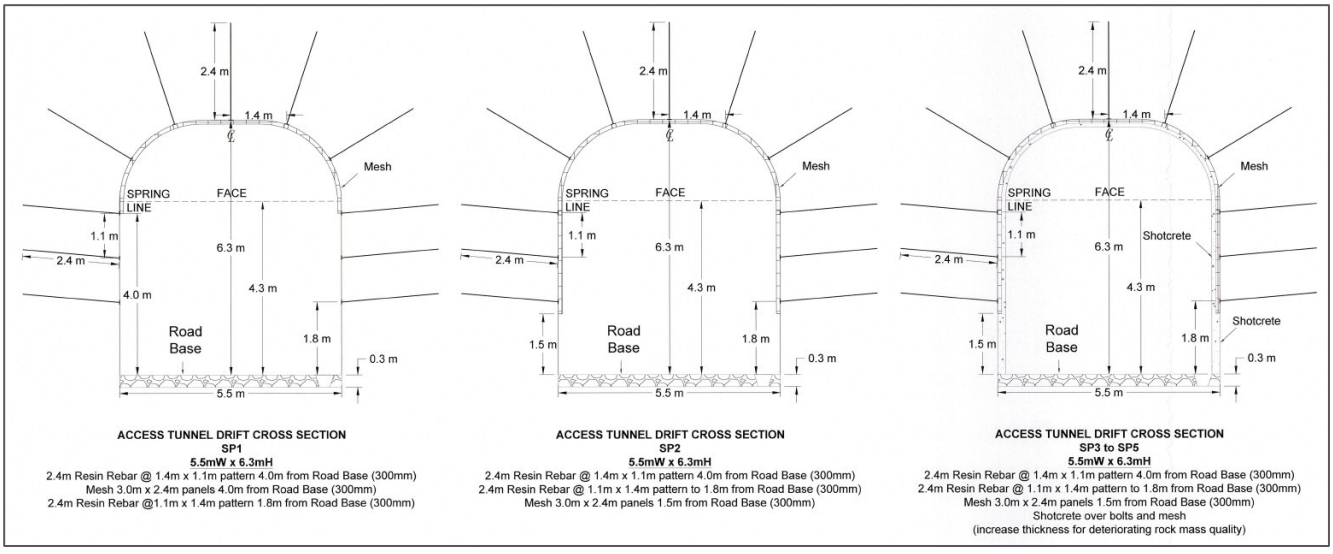
35 m Level spacing		Maximum side wall span (north to south)		
		Black Lake	Takla – Potassic	Takla – Phyllic
Expected/Typical Ground Conditions (85% Stopes)	450 m	29	23	18
	550 m	26	18	15
	670 m	23	16	No stopes
Lower Bound Ground Conditions (8% Stopes)	450 m	15	15	10
	550 m	14	13	8
	670 m	13	11	No stopes
Poor Ground Conditions (7% Stopes)	450 m	13	No stopes	8
	550 m	11	No stopes	No stopes
	670 m	10	No stopes	No stopes

Ground Support

Typical excavation dimensions were used to develop ground support assumptions and cost estimates for the preliminary economic assessment. Previous work completed by AMC (2017) was used to estimate ground support needs, associated costs, and development rates for this PEA. Another input carried from this work was the estimate that 15% of development will require shotcrete.

An example cross section with the associated ground support basis is shown by Figure 16-7.

Figure 16-7: Typical drift cross-sections showing rock support designs

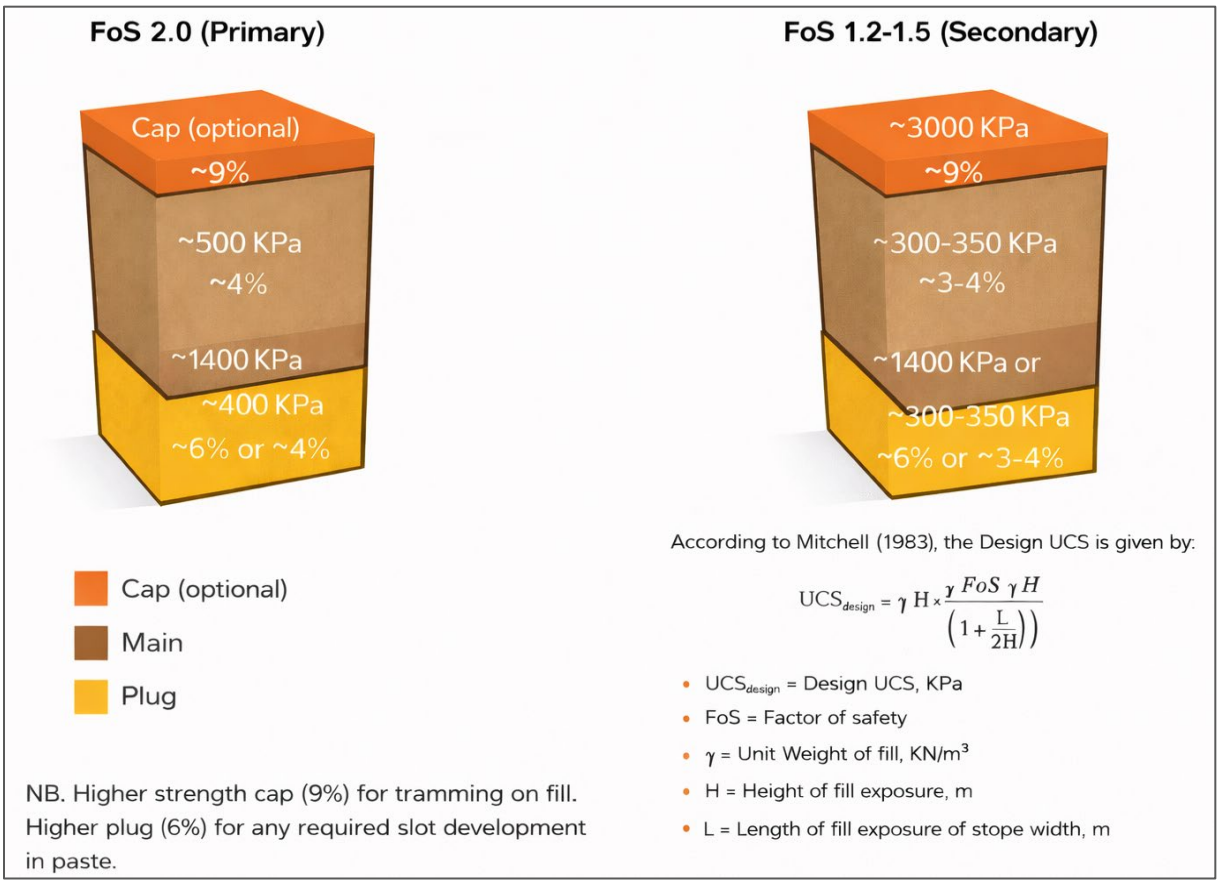


Backfill

Cemented paste from whole tailings have been selected as the backfill method for the KUG project. This backfill method was selected to improve recovery of mineralized material and ground stability. Faster stope cycling is possible compared to cemented rockfill and unconsolidated fill methods.

Preliminary backfilled stope strengths have been estimated using the Mitchell (1983)'s UCS formula. The results are shown in Figure 16-8, where various cement content percentages are specified for the primary and secondary stopes.

Figure 16-8: Paste backfill design criteria



The average binder content used for the PEA is 5.0% in the primary stopes and 3.3% in the secondary stopes. The paste fill batch plant is planned to be constructed at the mine portal location, and a booster station will be required in the decline to pump the paste to the stopes Refer to Item 18 for more details.

Stand-off and Pillar Sizes

Development was positioned to allow for a minimum 2:1 pillar size (minimum lateral offset is 2 x height of opening) where the pillar width was typically a minimum of 10 m. For the positioning of ramp infrastructure, a minimum offset of 50 m from the stoping zone was considered.

16.3 OPEN PIT MINE DESIGN AND PLAN

The Main Zone Open Pit will provide most of the feed to the mill. At peak production, the Main Zone will provide 86% of the mill feed with the balance coming from KUG. Mineralized material will be hauled to a surface crusher located on the east side of the open pit crest. Waste material will be hauled to a facility located to the northeast of the open pit.

Pioneering and early works to manage hazards associated with the talus slope are expected to require approximately two years prior to the commencement of production.

16.3.1 Importation of the Geological Model

The 2025 resource model was created by Centerra personnel in Leapfrog™ and later converted to Deswik format.

Framework details of the open pit block model are provided in Table 16-9. The final open pit mining model item descriptions are shown in Table 16-10. Deswik was used for pit optimization, using the Pseudoflow algorithm, and Deswik was used for phase designs and mine scheduling.

Table 16-9: Open pit block model prototype information

Framework description	Open pit model (value)
X origin (m)	634200
Y origin (m)	6324850
Z origin (m) (maximum)	-10
Rotation (degrees clockwise)	0
Number of blocks in X direction	286
Number of blocks in Y direction	166
Number of blocks in Z direction	134
X block size (m)	15
Y block size (m)	15
Z block size (m)	15

Table 16-10*:* Block model field definitions

Field item	Unit	Description	Range
Au	g/t	Gold grade	0-3.23
Cu	%	Copper grade	0-0.89
Ag	g/t	Silver grade	0-5.73
TOPO%	%	Proportion of block beneath topography	0-1
SG	g/cm^3		1.67-3.00
TF	cm^3/g	1/SG	0-0.4
CLASS	-	**Classification**: 1 – Measured; 2 – Indicated; 3 – Inferred; 4 – Potential; 0 – Unclassified	0-4
LITH	-	**Lithology codes**: 10000 – Air; 10100 – Overburden; 10200 – Hazelton; 10300 – Post mineralized dykes; 10400 – Soverign; 10500 – AAP Sill; 10600 – Black Lake; 10700 – Takla; 00000 – Unknown	10000 series
ALT	-	**Alteration codes**: 20000 – Air; 20100 – Overburden; 20200 – Leach-QSP; 20300 – Propylitic; 20400 – Advanced Argillic; 20500 – Potassic C-S Overprint; 20600 – C-S; 20700 – Phyllic; 20800 – Potassic; 20900 – Unaltered; 00000 – Unknown	20000 series
Broken_Zone	-	0 – Unknown; 1 – Broken Zone	0-1
UG_Model	-	1 – KUG; 2 – Offset; 3 – KE	1-3
PNAG	-	0 – Air/OB; 1 – PAG; 2 – PNAG; 3 – NAG; 4 – Unknown	0-4
NSR	CAD $/t	Net smelter return at US$2000/oz Au, US$4.00/lb Cu, and US$0/oz Ag	0-234.65
OSA	°	Slope Angle Input into Pseudoflow Optimisation	30-45

16.3.2 Open Pit Dilution and Recovery

No additional mining dilution or mining loss has been applied to open pit material in the PEA, as the block model and the style of mineralization is considered to incorporate dilution effects. This assumption has not been validated through detailed mine design and may change in subsequent studies.

16.3.3 Open Pit Process Recovery

Metallurgical recovery assumptions for the open pit are presented in Table 16-11. For the purposes of pit shell optimisation, leach circuit recoveries were not considered. Further information on this input can be referenced in Item 13. The recoveries for the open pit are organized by geometallurgical domain classification.

Table 16-11: Main Zone estimate of mineral recovery

Metallurgical domain	Gold (%)	Copper (%)	Silver (%)
Main Zone Pit (exclusive of broken zone)	60.3	91.1	45
Broken Zone	53.3	83.2	45

16.3.4 Pit Shell Development

Pit shells were generated using the Pseudoflow algorithm in Deswik software.

Table 16-12 summarizes the inputs into the revenue and cost calculations for the mine design application.

Table 16-12: Pit shell optimization inputs

Description	Units	Value
Resource model		
Block classification used	class	Indicated + Inferred (I + Inf)
Model block height	m	15
Mining bench height	m	15
Metal prices		
Gold price	US$/oz	2,000
Copper price	US$/lb	4.00
Silver price	US$/oz	0
Metallurgical information		
Process tonnes per operating hour	tph	2,083
Recovery – Gold	%	60.3
Recovery – Copper	%	91.1
Recovery – Silver	%	45
Recovery - Gold (Broken Zone material)	%	53.3
Recovery – Copper (Broken Zone material)	%	83.2
Recovery – Silver (Broken Zone material)	%	45

Description	Units	Value
Mining cost		
Mining	CA$/t mined	3.50
Incremental cost – Talus removal	CA$/t mined	0.60
Mine sustaining capital	CA$/t mined	2.00
Incremental cost – above El. 1707.5	CA$/t/15 m bench	0.06
Incremental cost – below El 1707.5	CA$/t/15 m bench	0.06
Processing and G&A cost		
Processing cost	CA$/t mill feed	7.10
Leaching cost	CA$/t mill feed	2.50
Conveyance cost	CA$/t mill feed	0.70
G&A cost	CA$/t mill feed	3.67
Total process & G&A	**CA$/t mill feed**	**13.97**
Concentrate transport		
Moisture content	%	8
Transport charges	US$/t conc (wet)	200.39
Transport losses	%	0.5
Selling – Copper concentrate		
Concentrate grade	%	21
Copper refining charge	US$/t payable lb	0.07
Smelting charge	US$/t dry conc. tonne	75
Copper Payable	%	96.5
Selling – Gold		
Gold refining charge	US$/ payable oz	5
Gold payable	%	97.8
Selling – Silver		
Silver refining charge	US$/ payable oz	**0.5**
Silver payable	%	85
Other		
Exchange rate	CA$: US$	1.33
Discount rate	%	5

OSAs were applied to the various pit design sectors. Certain sectors contained adjustments for expected ramp positioning or geotechnical berms. Table 16-13 summarizes the pit slope designs for each pit wall sector.

Table 16-13: Pit slope angles by pit wall sector

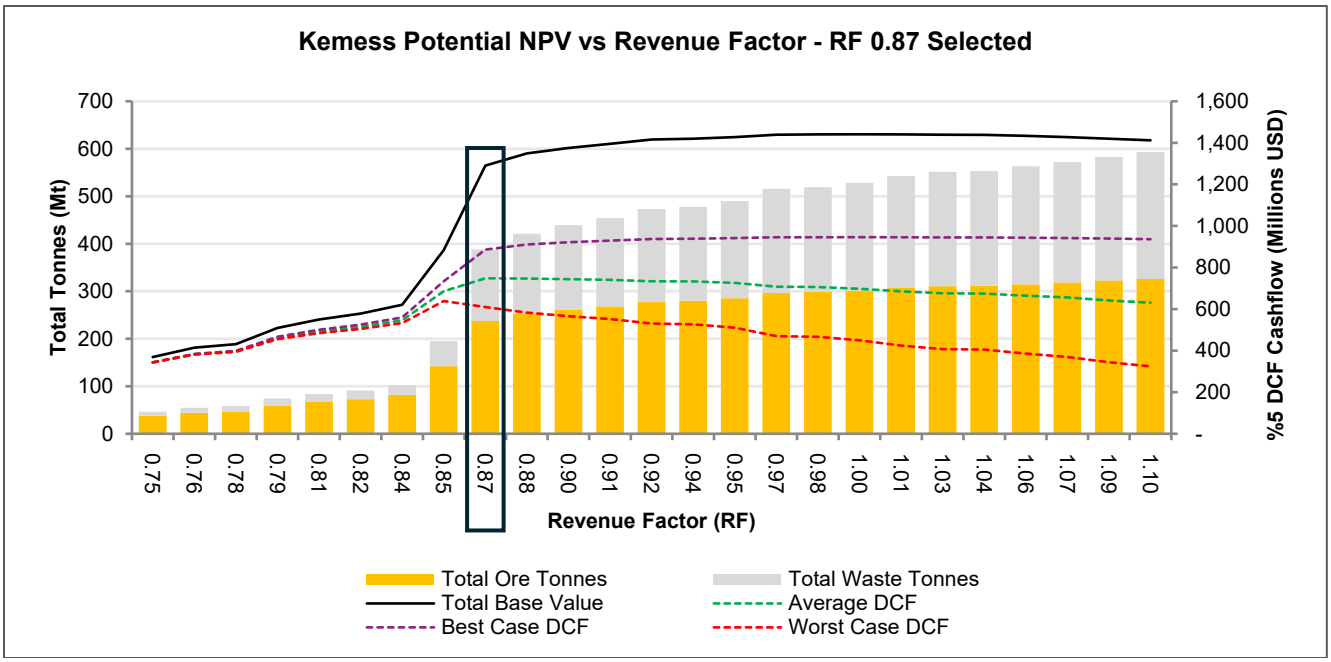
Pit design sector	Geotechnical domain	Design bench height (m)	Design face angle (°)	Design berm width (m)	OSA (°)	IRA (°)
All	Overburden	15	30	8	30	30
Main Pit West	Broken	15	60	8	42	42
Main Pit West	Takla	30	65	16	42	45
Main Pit East	Takla	30	70	15	43	49
Main Pit North	Takla	30	70	14	44	50
Main Pit North	Hazelton	30	70	14	45	50

Geotechnical berms were included every 150 m of slope height. Additional berms were included after pit shell generation as design factors for geohazard management around the talus area. The Douglas ridge crossing serves as both access and a geotechnical berm for the final phase.

A floor constraint to a maximum depth at 1,355 masl elevation was applied to the pseudoflow runs. This elevation represents the boundary between open pit and underground mining. MSO runs completed for KUG above this elevation (see section) did not result in any mining levels; therefore elevation 1,355 is considered as the optimum boundary between the two mining methods.

Pit shells were generated by varying revenue factor (RF) up to 1.10. The RF 0.87 shell ($1,740/oz gold and $3.26/lb copper) was selected as the final pit shell for design (Figure 16-9). The average discounted cashflow line approaches its maximum at this pit shell. It should be noted that the DCF only considers the potential revenue minus operating costs and excludes other costs including capital and sustaining. Other considerations for this selection included PAG storage facility size requirements and the projected mine life. Trade-off studies concluded that a three-phase strategy would provide consistent ore feed for the project. The RF 0.75 shell was chosen for phase 1 to provide approximately three years of ore feed. The second phase was chosen (RF 0.85 shell) to balance waste rock stripping requirements.

Figure 16-9: RF plot



16.3.5 Pit Designs

The phase designs include a pioneering phase (Phase 0) along with the three production phases. The pioneering phase establishes a ridgeline crossing between the west (and the road access to Kemess South) and the eastern half of the Main Zone Area. This crossing is required in early works to establish

access and facilitate construction of infrastructure, which includes the open pit crusher, planned to the east of the ridge. This crossing is used over the life of the mine for equipment and personnel access.

The first ore phase (Phase 1) targets three years of low-strip ore feed and uses the Douglas ridge crossing as the haulage route to the crusher. Phase 2 brings the pit walls to final design in the western half of the pit, and moves the pit exit to the east. Phase 3, the largest phase, is scheduled to begin stripping four years prior to its main ore release.

The pit design considers 15 m single benches in the overburden and broken zone geotechnical domains. Otherwise, final benches are designed to a height of 30 m but will be excavated in 15 m cuts.

Haulage ramp widths are designed to 32 m double lane and 20 m single lane. The design criteria are based on 3.5x truck widths and ¾ tire height berm width requirements. While mining regulations in British Columbia only require 3x truck width, the 3.5x running width ratio was chosen in consideration of snowfall and potential impacts on operating conditions. The design gradient of haulage ramps is 10%. Runaway lanes and/or retardation barriers will be considered where conditions warrant.

Phase design physicals are included in Table 16-14. Figures for each of the listed phases follow. Phase 0 is included in each figure as it forms the life of mine access around the north side of the planned open pit.

Table 16-14: Mining physicals by pit operations phase

Phase	Mill feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Gold conc. (koz)	Copper conc. (Mlb)	Silver conc. (koz)	Waste tonnes (Mt)	Total mined (Mt)	Overall pit strip ratio
Phase 0	-	-	-	-	-	-	-	22	22	-
Phase 1	33	0.33	0.18	1.13	354	128	1193	10	43	0.30
Phase 2	91	0.29	0.14	1.03	856	282	3031	63	155	0.69
Phase 3	100	0.32	0.16	1.13	1029	340	3625	73	173	0.73

Figure 16-10 illustrates the pre-production mining phase and Figure 16-11 through Figure 16-13 show the extent of excavation after each production mining phase.

Figure 16-10: Phase 0 (Ridgeline Crossing)

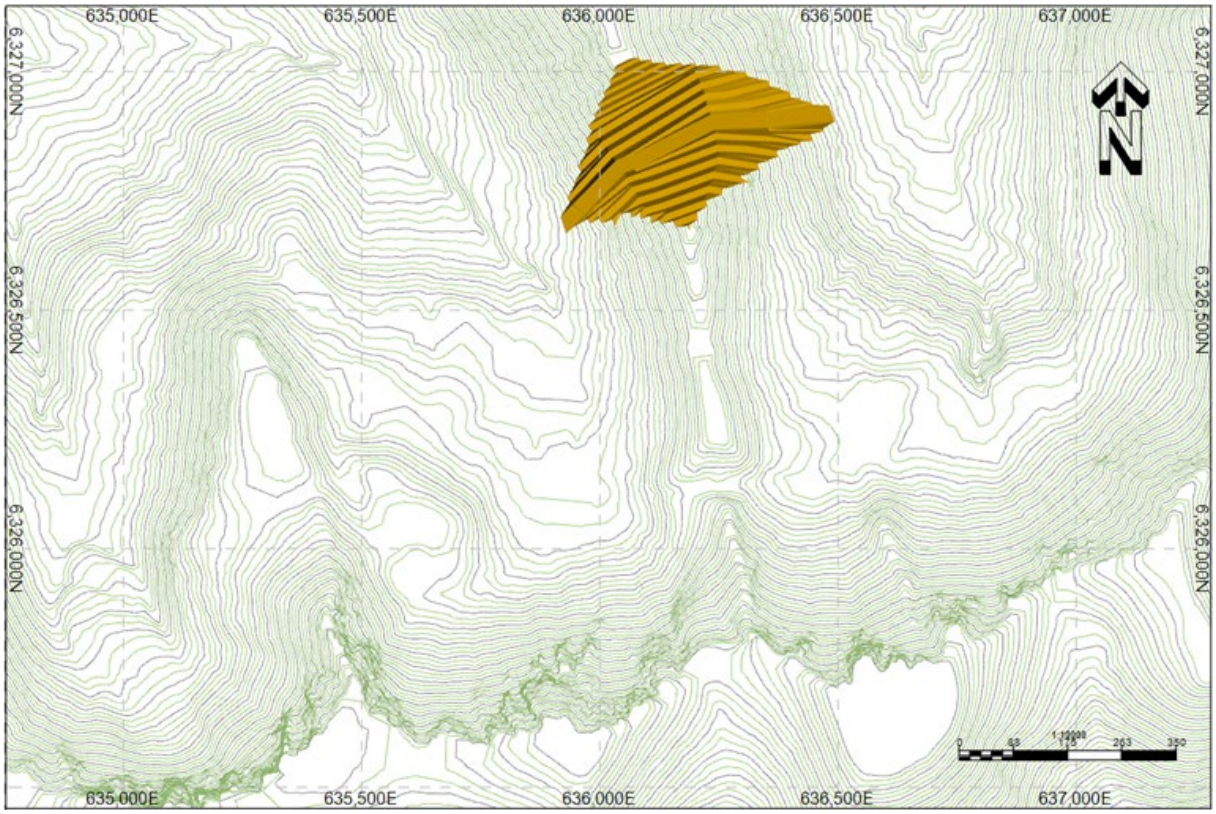


Figure 16-11: Phase 1 Mine Plan

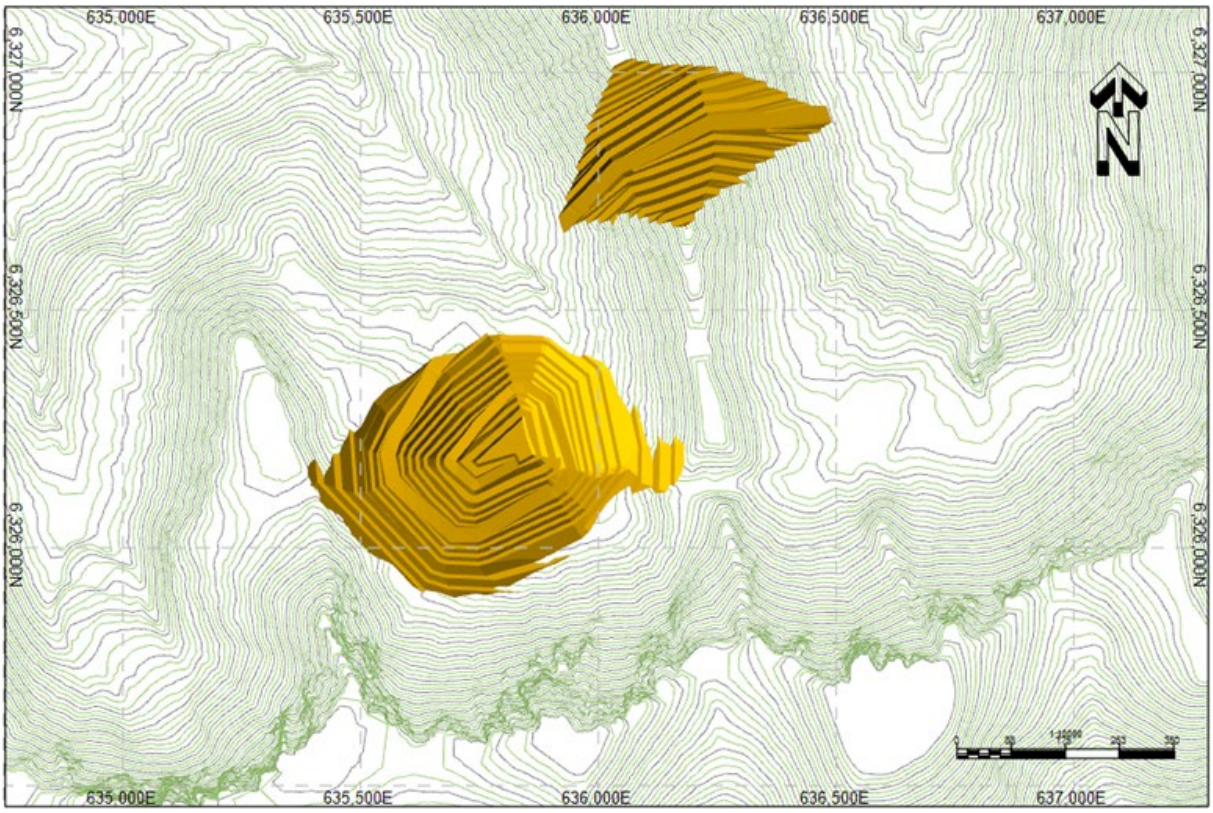


Figure 16-12: Phase 2 Mine Plan

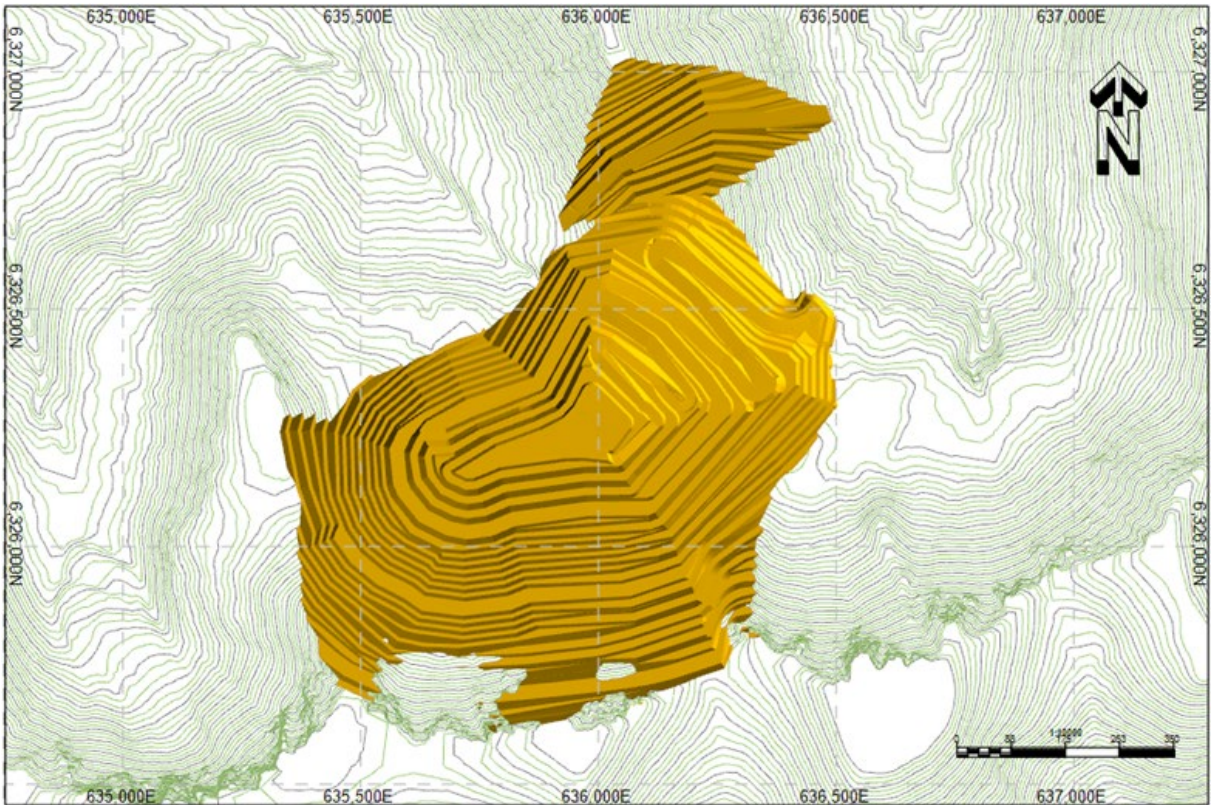
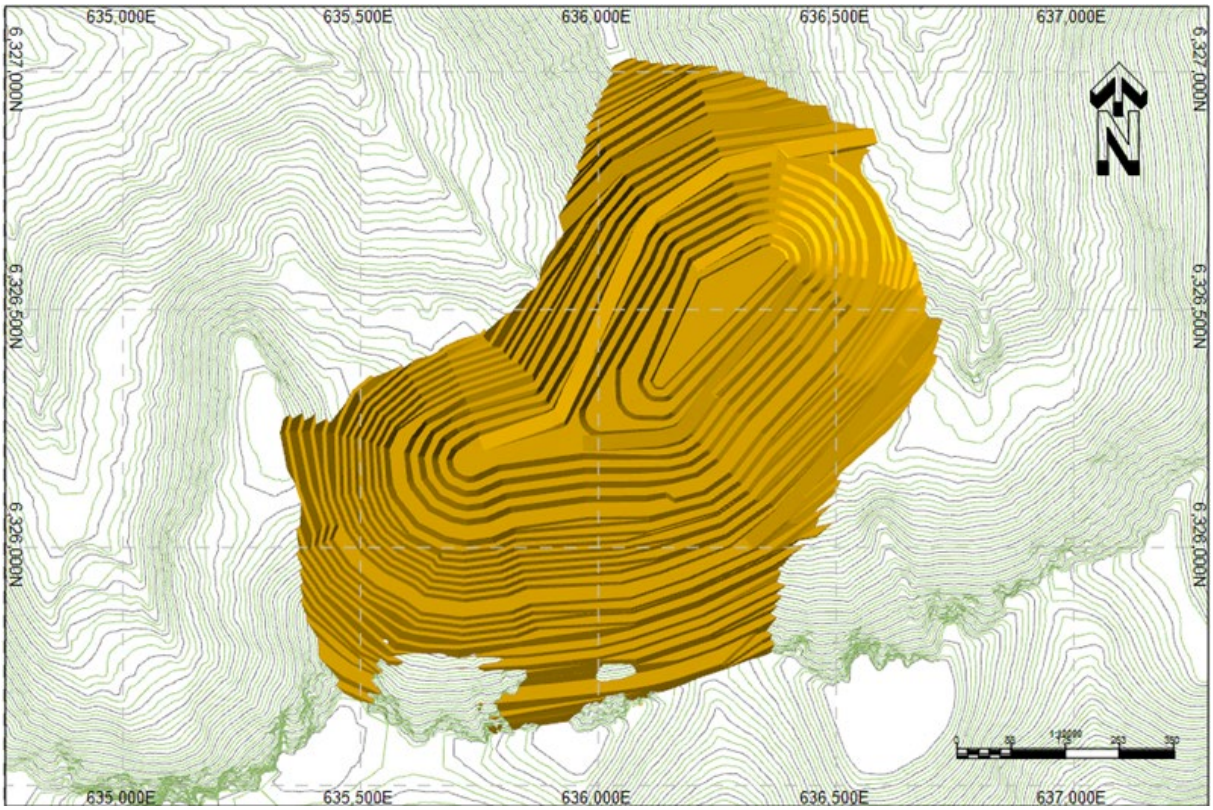


Figure 16-13: Phase 3 Mine Plan

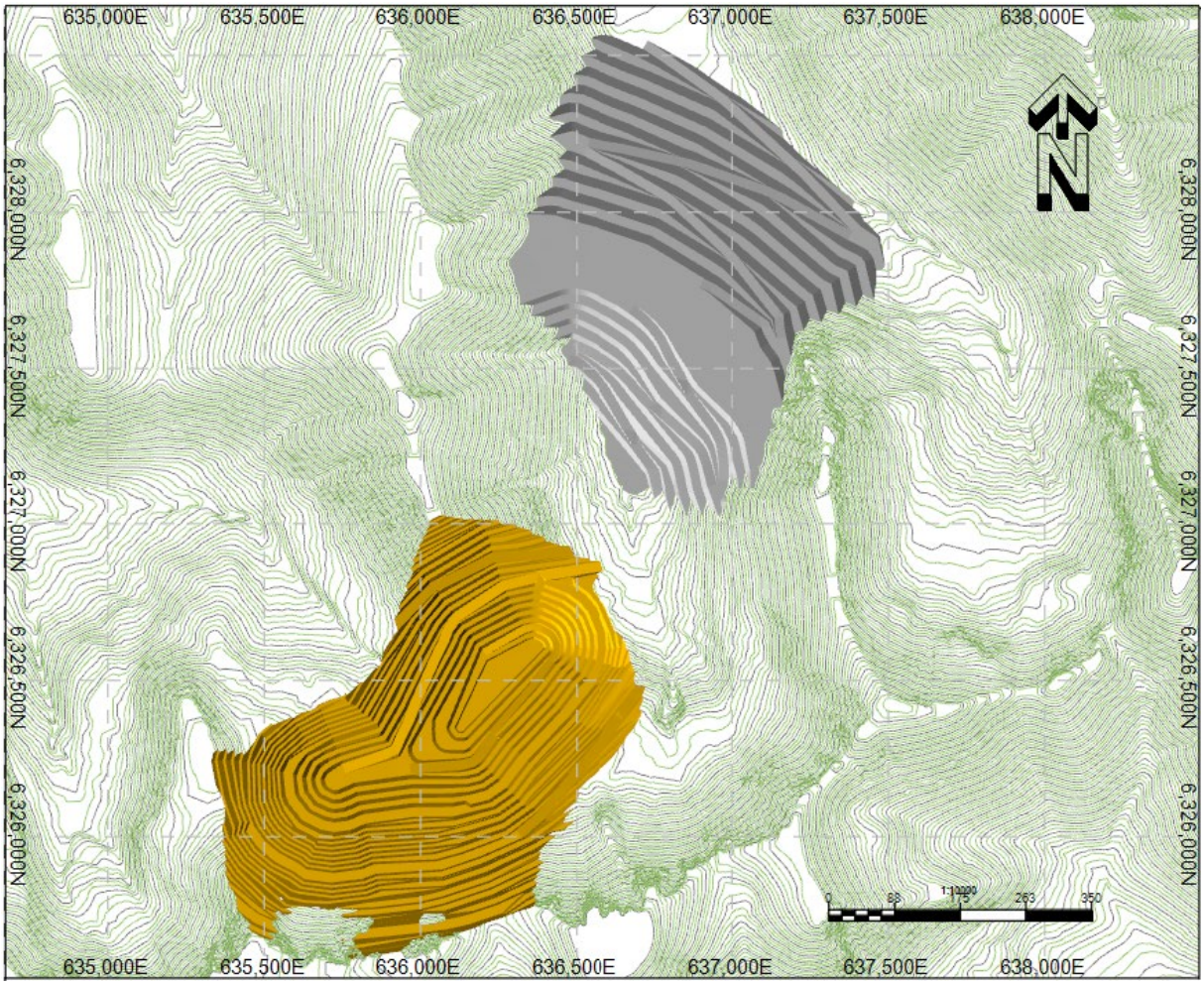


16.3.6 Waste Rock Storage and Ore Stockpiles

A waste rock storage facility (WRSF) will be required for open pit mining. The facility is located to the north of the Main Zone Pit. The designed volume of the facility is 100 Mm^3, which includes 20% contingency storage capacity. As discussed in Item 20.1.3, 99% of the waste rock is estimated to be PAG. The mine plan stores all waste rock in this facility. For the purposes of the PEA, the construction of this facility considers overburden removal, an engineered liner, and a construction fill material base. In addition, the PEA considers seepage collection, trenching, and water runoff management. A map of this facility is included in Figure 16-14.

The design criteria for the WRSF facility includes an overall slope of 2H:1V. A single lane ramp is included on the north side for environmental monitoring access. Short-term stockpiling of mineralized material will be within the footprint of the WRSF this facility. Figure 16-14 shows the planned location of the WRSF relative to the ultimate pit limits.

Figure 16-14: Main Zone Pit and PAG facility layout



16.3.7 Open Pit Mining Schedule

The Open Pit Mining Schedule includes calendar adjustments for inclement weather, equipment availability, and other operational delays. A summary of these factors is included in Table 16-15.

Table 16-15: Open Pit Mining Schedule inputs

Item	Unit	Quantity
Weather days	days	17
Non-operating time allotted to shift changes	days	3
Remaining days for scheduling	**days**	**345**
Average equipment utilisation	%	79
Average equipment availability	%	88

The open pit mining sequence begins with one pre-production phase (“Phase 0”) to establish access across Douglas ridge in Year -2 and Year -1. In addition, “Phase 0” was designed to provide the NAG material required for infrastructure construction. During the pre-construction period, the removal of necessary talus material along the south slope is also completed. Mining rates for these activities are reduced for small scale equipment. The talus mining specifically is scheduled for dayshift and summer season only.

The first production phase is projected to be Phase 1 (Figure 16-11). Phase 1 has a relatively low stripping ratio and will begin shortly before Mill Ramp-Up in Q4 Year 1. Two million tonnes of Phase 1 material will be stockpiled for processing in Year 2. The stockpiling is assumed to be placed on the PAG facility footprint or at the mill. No further long-term stockpiling is required in the mine plan. The haulage profiles for Phase 1 use the Douglas Ridge crossing to access the crusher and waste rock storage facility.

Waste stripping for Phase 2 and Phase 3 (Figure 16-15 and Table 16-16) will commence while mining from Phase 1 progresses. Early stripping for the two phases is needed to meet mill feed requirements in later years of the mine plan.

Figure 16-15: Main Zone Pit Phase Mining Schedule

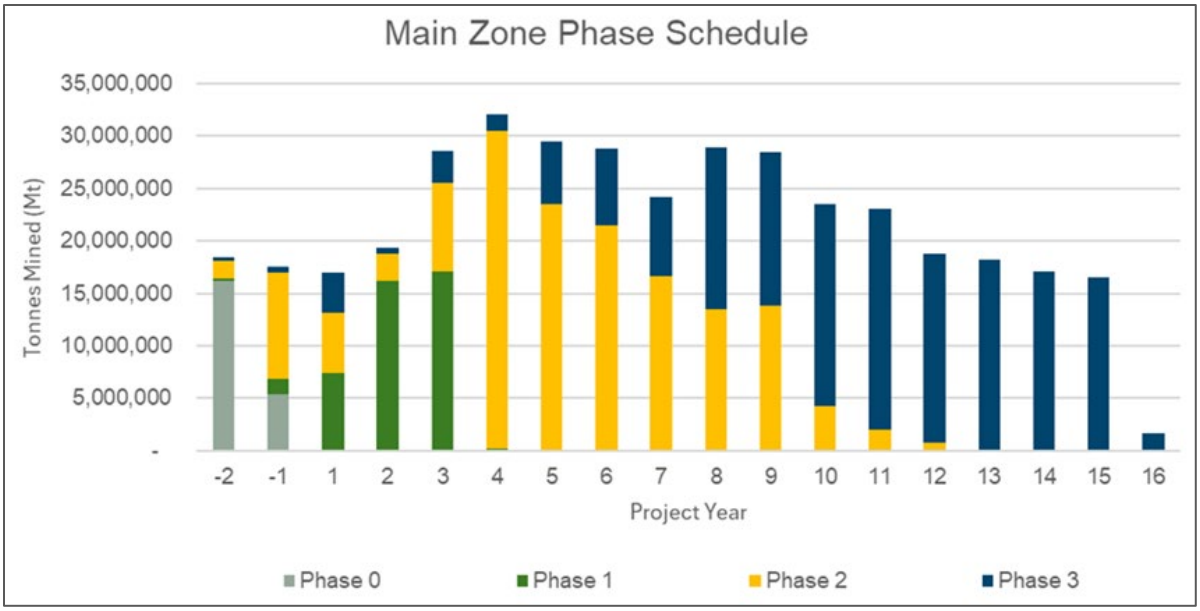


Table 16-16: Annual tonnes mined by pit phase

Phase	Total	Year																	
		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
0	**21.6**	16.2	5.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	**42.6**	0.3	1.4	7.4	16.2	17.1	0.2	-	-	-	-	-	-	-	-	-	-	-	-
2	**154.6**	1.7	10.1	5.7	2.5	8.5	30.2	23.6	21.5	16.6	13.5	13.8	4.2	2.0	0.7	-	-	-	-
3	**172.9**	0.3	0.6	3.9	0.6	3.0	1.6	5.9	7.3	7.5	15.4	14.6	19.3	21.0	18.0	18.3	17.1	16.6	1.7
Total	**391.7**	**18.4**	**17.6**	**17.0**	**19.3**	**28.6**	**32.0**	**29.5**	**28.8**	**24.2**	**28.9**	**28.4**	**23.5**	**23.0**	**18.7**	**18.3**	**17.1**	**16.6**	**1.7**

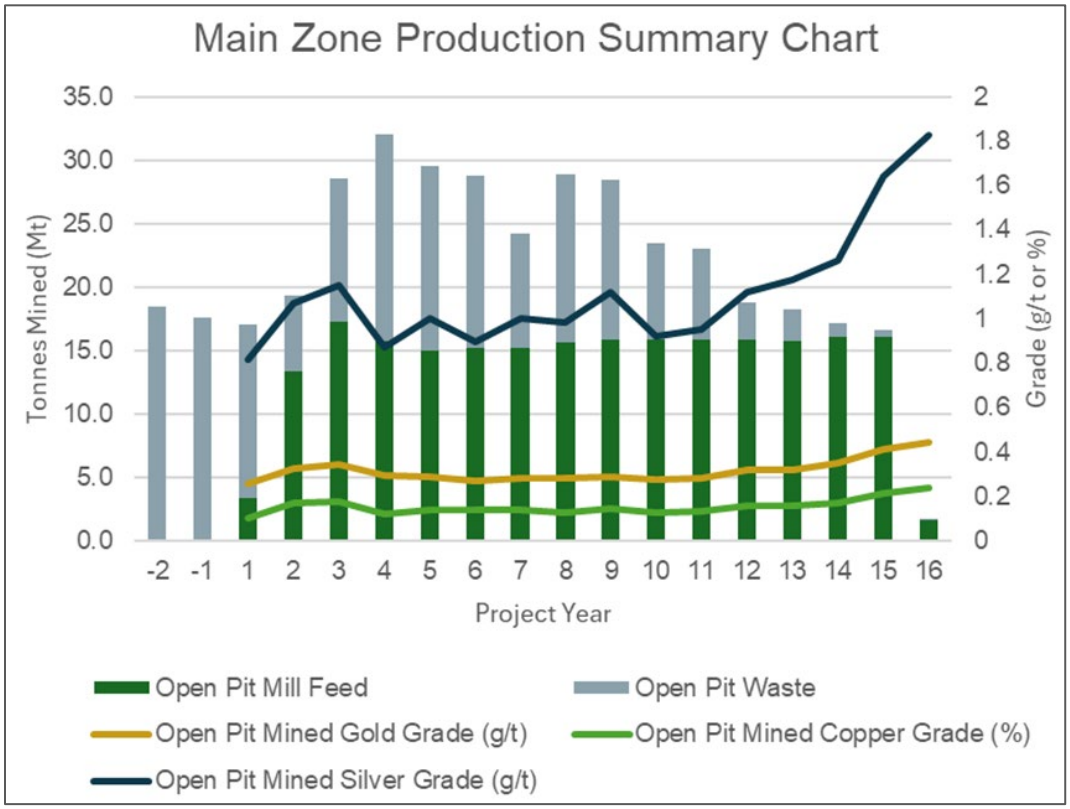
Table 16-17: Open pit mine production

Item	Units	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
ROM mined – directly to mill	Mt	**222**	-	-	1.4	13.4	17.3	15.7	14.9	15.2	15.2	15.7	15.9	15.8	15.8	15.9	15.7	16.1	16.1	1.7
ROM mined – to stockpile	Mt	**2**	-	-	2.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total mill feed	**Mt**	**224**	**-**	**-**	**3.4**	**13.4**	**17.3**	**15.7**	**14.9**	**15.2**	**15.2**	**15.7**	**15.9**	**15.8**	**15.8**	**15.9**	**15.7**	**16.1**	**16.1**	**1.7**
Waste mined	Mt	**168**	18.4	17.6	13.6	5.9	11.3	16.3	14.6	13.6	9.0	13.2	12.5	7.7	7.2	2.8	2.5	1.1	0.5	0.1
Total material mined	Mt	**392**	18.4	17.6	17.0	19.3	28.6	32.0	29.5	28.8	24.2	28.9	28.4	23.5	23.0	18.7	18.3	17.1	16.6	1.7
Rehandle ore to mill	Mt	**2**	-	-	-	2.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total material moved	**Mt**	**394**	**18.4**	**17.6**	**17.0**	**21.3**	**28.6**	**32.0**	**29.5**	**28.8**	**24.2**	**28.9**	**28.4**	**23.5**	**23.0**	**18.7**	**18.3**	**17.1**	**16.6**	**1.7**
Strip ratio	W:O	**0.8**	-	-	4.0	0.4	0.7	1.0	1.0	0.9	0.6	0.8	0.8	0.5	0.5	0.2	0.2	0.1	0.0	0.0
Au Grade mined	g/t	**0.31**	-	-	0.26	0.32	0.34	0.30	0.29	0.27	0.28	0.28	0.29	0.28	0.28	0.32	0.32	0.35	0.41	0.44
Cu Grade mined	%	**0.15**	-	-	0.10	0.17	0.18	0.12	0.14	0.14	0.14	0.13	0.15	0.13	0.13	0.16	0.16	0.17	0.21	0.24
Ag Grade mined	g/t	**1.09**	-	-	0.82	1.07	1.15	0.87	1.00	0.90	1.00	0.99	1.12	0.92	0.95	1.12	1.18	1.26	1.65	1.83
Contained Au mined	**koz**	**2,234**	**-**	**-**	**28.1**	**139.9**	**190.5**	**150.1**	**139.7**	**131.9**	**137.4**	**140.7**	**146.6**	**141.1**	**143.8**	**164.0**	**160.8**	**182.1**	**213.2**	**23.8**
Contained Cu mined	**Mlb**	**746**	**-**	**-**	**7.7**	**50.5**	**68.1**	**42.0**	**45.8**	**45.7**	**45.8**	**44.1**	**51.1**	**43.8**	**45.7**	**56.5**	**54.9**	**61.0**	**75.0**	**8.7**
Contained Ag mined	**koz**	**7,829**	**-**	**-**	**88.7**	**461.7**	**643.1**	**439.8**	**482.2**	**437.4**	**488.0**	**496.3**	**571.7**	**467.4**	**484.3**	**574.5**	**594.9**	**651.4**	**849.3**	**98.2**

Note: Figures may not sum precisely due to rounding.

A summary chart of the planned open pit production is depicted with Figure 16-16. The gold and copper grade profiles are relatively consistent throughout the mine plan. A small increase in planned grades is observed at the end of the mine plan, when the open pit depth is approaching the higher-grade underground mine.

Figure 16-16: Open pit mined tonnes and metal grades



16.3.8 Open Pit Mining Equipment

The loading fleet strategy considers one electric shovel (30 m^3 bucket capacity), one hydraulic shovel (29 m^3), and one wheel loader (21 m^3). One-hundred-forty tonne class haul trucks were selected to maximize loader productivity and operational flexibility. Support equipment identified for mining the talus in the pre-production period and other tasks are included in the equipment list on Table 16-18. Equipment requirements were determined using a combination of first-principles estimates and benchmarked production rates.

Table 16-18: Open pit mining equipment list

Open pit mine production equipment		
Model (or equivalent)	**Description**	**Quantity**
Caterpillar 785	Haul Truck	16
P&H 2800	Electric Shovel	1
Komatsu PC 5500	Hydraulic Shovel	1
Caterpillar 995	Wheel Loader	1
Epiroc PV271	Rotary Drill	2
Epiroc D65	DTH Drill	1
Caterpillar D10	Dozer	3
Talus and Pioneering support equipment		
Caterpillar 6020	Excavator	1
Caterpillar 390	Excavator	2
Caterpillar 740	articulated truck	2
Caterpillar D5	Dozer	1
Support equipment		
Water Truck		1
Graders		2
Rubber Tire Dozer		1
Tire Handler		1
Emulsion Truck		2
Light Vehicles		7
Crew Bus		2
Lube Truck		1
Nipper Service Truck/Mechanic Truck		3
Loaders for Crusher and Stemming		3

16.3.9 Open Pit Dewatering

Dewatering requirements for the open pit will include in-pit pumps and diversion ditches to control surface runoff. Vertical wells and horizontal drains are not identified as being required at this time. A collection pond will be required for managing contact water from both the PAG stockpile and the open pit. Further information on water modelling can be referenced in Item 18.6.

16.3.10 Open Pit Mine Infrastructure

Haul roads outside the pit perimeter are designed to a 27 m running width to accommodate the 140-t class haul truck traffic.

The mine plan leverages the existing facilities at Kemess South where possible (see Section 18). Larger rebuilds/equipment repairs will be performed at the KS Truck Shop. However, several new facilities will be required within proximity of the Main Zone Pit. This infrastructure is planned to include:

- Fuel storage

- Explosives and detonator storage
- Preventative maintenance facilities
- Offices and lunchroom.

16.3.11 Open Pit Conveyor Tunnel

A 3.9 km inclined tunnel at 4% grade for conveyance of crushed ore is planned from the south transfer station area to the surface crusher location. Construction of the tunnel will begin three years before open pit ore delivery to the processing facility is scheduled. A conveyor will be installed in the tunnel.

An underground bypass to access the open pit from the south was designed to allow access in addition to material handling via the conveyor. The primary conveyor tunnel was designed to include rehandle of muck, water collection sumps, ventilation raises, and allocations for installing crusher infrastructure and conveyors for material movement. The development schedule specific to the pit conveyor infrastructure is summarised in Table 16-19.

Table 16-19: Open pit conveyor drive schedule

Period	Capital lateral development (km)	Capital vertical development (km)	Waste material (kt)
Total	**5.5**	**0.4**	**848**
-3	1.5	0.1	265
-2	3.3	0.3	538
-1	0.3	0.0	45

Due to permitting, development from the South portal will begin at the start of construction and development from the North side portal will be delayed by one year.

16.4 UNDERGROUND MINE DESIGN AND PLAN

Previous studies on the KUG project have considered block caving as the mining method for the deposit. Internal trade-off studies between longhole stoping with paste backfill and sublevel caving have been conducted as alternative mining methods to block caving to reduce upfront capital and improve overall mined grade. For the current PEA, longhole stoping with paste backfill was selected which is expected to meet the economic, geotechnical and productivity objectives of Project management.

Longhole stoping considers both transverse and longitudinal stoping, with the transverse technique selected where the thickness of the mineralisation is consistently above 18–20 m and over 80 m in height. For Kemess, transverse stoping uses both primary and secondary stopes mined in a bottom-up sequence. Longitudinal stoping will be applied in narrow mineralisation and sequenced bottom-up. Mined mineralisation from production is proposed to be dropped into vertical passes that transfer

material to an underground crushing infrastructure. The material will then be conveyed to a surface transfer station and combined with the open pit material prior to conveyance to the processing facility.

AuRico engaged an independent consultant experienced in underground mining to provide an underground mine design for the PEA on the Kemess Project. Available data, studies and previous evaluations were gathered to inform its assessment of a potential underground longhole stoping mine operating simultaneously with the overlying open pit mine. The QP has reviewed the study and accepts its designs, schedules and conclusions for the integrated Kemess Project PEA.

16.4.1 Underground Mine Access

The KUG mine will be accessed from two declines with portals 3.6 km to the south-east from the mining area. One portal will be used for material handling and the other will be used for services, secondary egress, and personnel. The underground portals are located in proximity to the main conveyor tunnel portal and are planned to share ventilation synergies but not personnel access.

16.4.2 Proposed Mining Methods

Longhole stoping is the proposed mining method for KUG with both longitudinal and transverse stoping. Transverse stoping is recommended as the dominant stoping method with a primary and secondary stoping sequence. Only a minor amount (<2%) of the deposit will employ longitudinal stoping.

Sublevel vertical spacing is proposed to be 35 m, with a target stope strike length of 20 m. Strike lengths may vary depending on ore body continuity, geometry, and ground conditions. Stope widths are 20 m for transverse mining with alternating primary and secondary stopes. Longitudinal shapes are designed at a minimum of 7 m width, including 2.5 m of unplanned rock overbreak, and a maximum of 20 m wide.

All stopes will use paste fill to support excavated production stopes. Required infrastructure including a single module paste fill plant is planned. It is expected that the paste fill plant productivity will determine the overall production rate for the mine.

16.4.3 NSR and Cut-off Grade

The KUG project will produce a copper concentrate (containing copper, gold and silver) and doré (gold and silver). The following formulas were used to calculate the NSR of the mineable inventory. The NSR was calculated in the block model on a grade basis (CA$ NSR/t) for use in the Mineable Shape Optimisation (MSO) process. Figure 16-17 shows the formulae used to determine gold and copper recoveries for the purpose of NSR and cut-off grade calculations.

Figure 16-17: Gold and copper recovery formulae

Copper Recovery %: $\%\ Rec_{Cu} = 101.26\ \%\ Cu_{Head}{}^{0.0733}$ eq 1

Gold Recovery %: $\%\ Rec_{Au} = 7.3536\ Au_{Head,g/t} + 63.261$ eq 2

Where $\%\ Cu_{Head}$ refers to the amount of copper (%) in the head grade (mill feed) and $Au_{Head,g/t}$ refers to the amount of gold (g/t) in the head grade (mill feed).

Notes: (1) Maximum copper recovery is 96%. (2) Maximum gold recovery is 84.3%.

Net smelter return:

$$\left[NSR\frac{CA\$}{t}\right] = ([\text{Total Revenue}] - [\text{Total Freight Cost}] - [\text{Total Smelter Charge}]) \times [\text{Exchange Rate}])$$

A cut-off grade is used to segregate material by whether the estimated revenue in a block exceeds the estimated costs of extraction and processing of that block. There were three cut-off grades used to assess and schedule mining at KUG: Fully Costed Cut-Off Grade (FCCOG), Incremental Cut-Off Grade (ICOG) and Marginal Cut-Off Grade (MCOG). Material within the block model with resource classification of inferred or indicated was considered for the cut-off grade. All other material classifications were assigned no value nor recovery in the NSR and cut-off grade evaluation.

A summary of the included costs in calculating cut-off grade are summarised in Table 16-20.

Table 16-20: Cost included in mine cut-off grade

Cut-off value	G&A	Processing	Surface handling	Mining	Sustaining capital	Operating development
Fully costed	✓	✓	✓	✓	✓	✓
Incremental	✓	✓	✓	✓	✓[1]	✗
Marginal	✓	✓	✓	✗	✗	✗

1 – Partially Included (UG fleet and fixed plant replacement included).

The cut-off grade was based on collected cost data and summarised in Table 16-21.

Table 16-21: Preliminary estimates of the various cut-off values

Parameter	Unit	LHS
Marginal costs	CA$/t	$14.60
Incremental costs[1]	CA$/t	$54.10
Fully costed costs	CA$/t	$56.10

1 – Used for MSO Stope Evaluation.

16.4.4 Dilution and Mining Recovery

Mining factors are used to account for the combination of dilution and recovery that affects the material quality and quantity to be mined.

For preliminary design at the KUG mine, planned dilution and unplanned rock dilution was estimated using the Datamine Mineable Stope Optimiser® software (MSO). Unplanned fill dilution and mining loss was applied as a factor to the shapes created by MSO within the schedule. The factors were estimated using sidewall geometry, stope sequence, and the ELOS method (Equivalent Linear Overbreak/Slough, Capes and Milne 2008).

For mineralized development material, mining recovery was estimated to be 100% with 10% unplanned dilution. Mining recovery for longhole stoping was assumed to be 94% for primary stopes, 92% for secondary stopes, and 93% for longitudinal stopes.

Table 16-22 summarises the dilution and mining recovery assumptions. Unless dilution was interrogated during the MSO process supported by the values in the block model, dilution was assigned zero grade.

Table 16-22: Dilution and recovery summary for KUG Project

Description	Units	Primary stoping	Secondary stoping	Longitudinal stoping	Development
Total planned dilution[1]	**%**	**9.6**	**9.3**	**31.7**	**-**
Total unplanned dilution	**%**	**7.5**	**17.3**	**14.7**	**10**
Unplanned fill dilution[2]	%	5.3	15.3	4.1	-
Far Wall	*m*	*1*	*1*	*1*	-
Floor	*m*	*0.25*	*0.25*	*0.25*	-
Sidewall – HW / FW	*m*	-	*1 / 1*	-	-
Unplanned rock dilution[3]	%	2.2	2.0	10.6	10
Minimum target mining width	m	20	20	4.5	-
ELOS – HW	*m*	-	-	*1.25*	-
ELOS – FW	*m*	-	-	*1.25*	-
Minimum total mining width	m	20	20	7.0	-
Mining recovery[4]	**%**	**94**	**92**	**93**	**100**

1. Included in MSO shape
   - Weighted average by target mineralisation tonnes within mining shape
   - Expressed as total planned material below cut-off divided by target mineralisation above cut-off.
2. Applied as factor to volume of the shape (assumed density of backfill = 2.00 t/m^3).
3. Included in MSO shape (stoping only)
   - Weighted average by total tonnes within mining shape
   - Included in MSO shape as interrogated dilution
   - Expressed as total unplanned material tonnes divided by target shape tonnes.
4. Applied as a factor to the final shape tonnes as a reduction in planned and unplanned material.

Stope design parameters were based on typical values for similar mining methods. The parameters for stope generation in MSO are listed in Table 16-23.

Table 16-23: Preliminary MSO parameters

Parameter	Unit	Longhole
ICOG	CA$/t NSR	54.10
Fill dilution estimate	%	5.7
Fully costed Cut-off Value	CA$/t NSR	56.10
Minimum mining width	m	4.5
ELOS – FW[1]	m	1.25
ELOS – HW[1]	m	1.25
Final minimum mining width	m	7.0
Stope assessment length	m	20
Target final Stope length	m	20
Stope height	m	35

1 – Transverse shapes have FW and HW ELOS applied at extents only.

The generation of stope shapes was constrained to a top elevation of 1,355 masl for the PEA Mine Plan. This elevation was determined as the ideal change-over point between open pit and underground mining during preliminary MSO runs. The KUG deposit has a high-grade core and additional mining levels are not economic above this elevation at the current input prices.

Following generation of the stope shapes, a preliminary economic screening was completed to test whether the selected shapes would pay for the estimated development required for access. In this preliminary evaluation process additional non-financial restrictions were also considered, such as geotechnical pillars, topographical standoffs, intersection with the open pit mining zone and mining practicality.

The parameters used to complete the preliminary economic screening are summarised in Table 16-24.

Table 16-24: Key economic screening parameters

Parameter	Unit	Value
Development – Capital – Lateral	CA$/m	10,000
Development – Operating – Lateral	CA$/m	8,000
Development – Capital – Vertical	CA$/m	12,000
Incremental Costs - Longhole	CA$/t mined	54.10
Revenue Factor	CA$/NSR	1.00

16.4.5 Underground Mine Design

Development

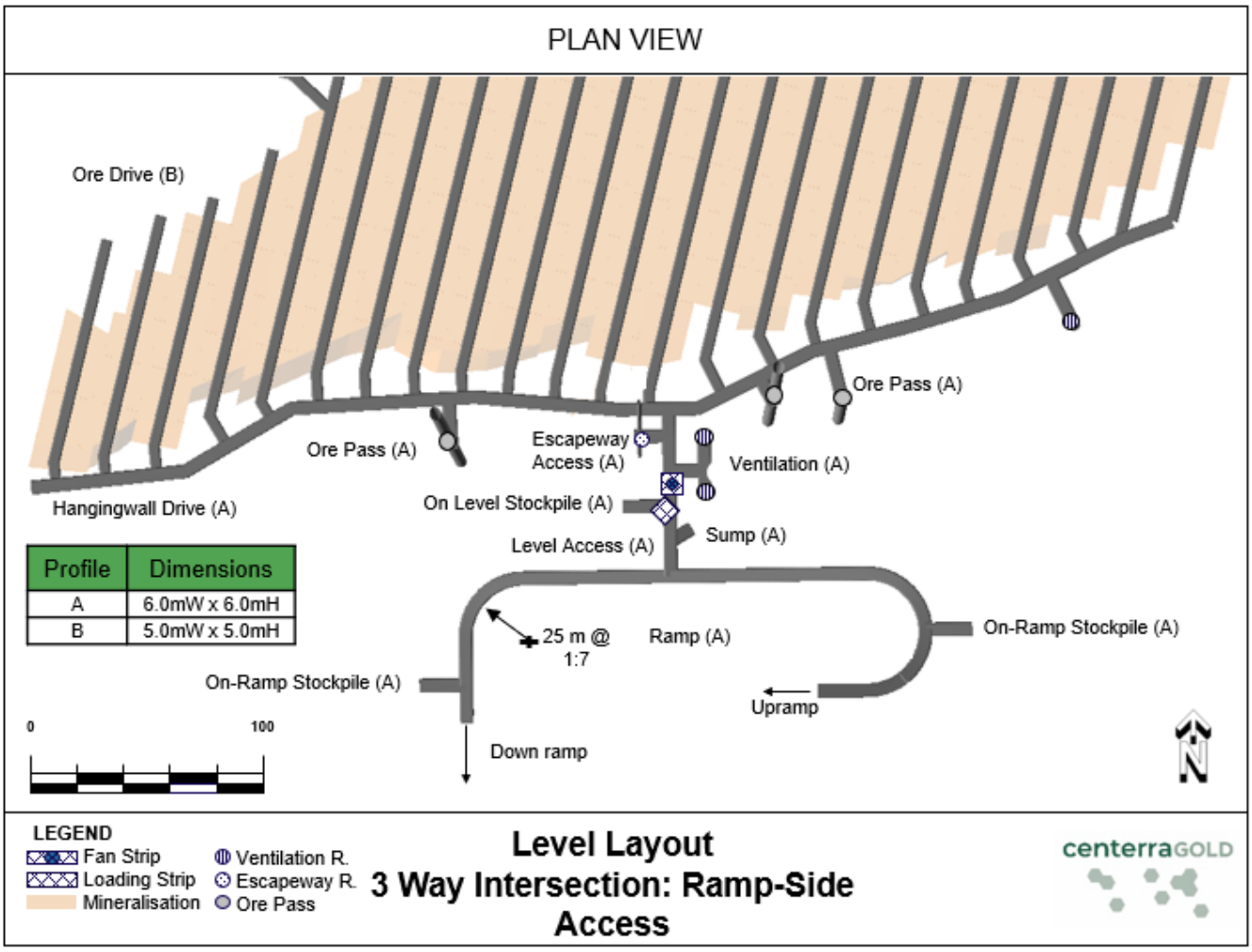
The development design incorporates a minimum stand-off distance of 50 m from ramp to mineralized bodies. This distance is included to avoid damage to the ramp due to ground stress changes and blasting from stope extraction. This stand-off distance also allows sufficient space between the ramp and the mineralised body for the excavation of the level accesses, stockpile cutouts, and sumps.

The ramp will be excavated to a width of 6 m and a height of 6 m with an arched profile. This profile allows sufficient room to accommodate 60–63 t underground trucks, as well as secondary ventilation ducting and service piping required for advancing development and maintaining production. Other planned development includes the following:

- Access drifts
- Sills (development on mineralisation)
- Operating waste development (sills mining material below cut-off)
- Sumps, escapeways, and accesses to the escapeways
- Return airways and accesses to the return airways
- Stockpiles
- Footwall drives.

A typical level layout for the mine is provided in Figure 16-18.

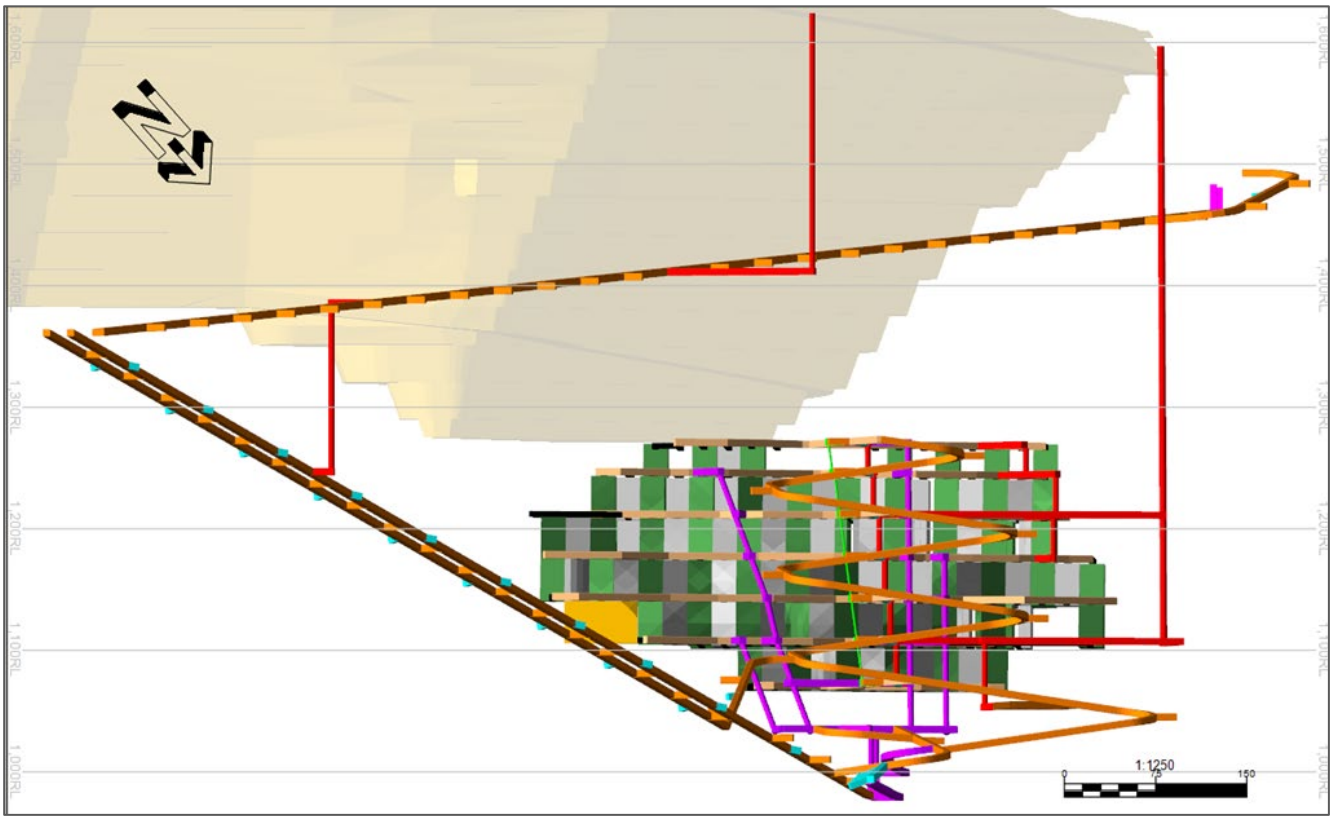
Figure 16-18: KUG typical development level layout



The main ramp access from the portal area will be via two declines with one access for equipment and personnel and the other for installation of a conveyor system. The ramps will diverge just before reaching the deposit, with the conveyor drift extended below the deposit and the access ramp connecting to each production sublevel.

The proposed design allows for loaders to tram material from stope draw points directly to passes positioned in the hangingwall drive, where material will be transferred to underground crushing infrastructure and transferred by conveyor belt to the processing facility. A surface transfer station at the portal is proposed to transition the underground material to the main conveyor system. The overall mine development is illustrated in Figure16-19.

Figure16-19: Isometric view of the proposed KUG mine



Underground Mine Production

Following MSO analyses, the shapes identified as transverse stoping were sliced into 20 m lengths and any longitudinal shapes more than 20–25 m wide were also split (longitudinally) for mine planning purposes. The stopes were sequenced bottom-up, retreating towards the decline access. The primary and secondary transverse shapes are sequenced so that secondary stopes can advance alternately with primary stopes so long as the primary stopes have a two-stope offset. The grade is highest towards the center of the deposit; therefore, those shapes were prioritized. The resulting sequence created a 3D echelon retreat. The sequence and stope size parameters are summarized in Table 16-25.

Table 16-25: Stope dimensions by mining method

Description	Units	Transverse stoping	Longitudinal stoping
Stope height	m	35	35
Stope length	m	10–20	10–20
Stope width	m	20	7–20
Mining direction	-	Bottom-up	Bottom-up
Retreat direction	-	North:South	East:West / West:East

Orphan Analysis

An orphan analysis is a final direct-cost economic assessment of the designed mine plan to determine the economic extents to be included for scheduling. The orphan analysis was completed using Deswik's Pseudoflow Tool. Pseudoflow uses the Lerchs-Grossman optimisation algorithm to identify the required revenue factor of an activity (or set of activities) to be included in the plan. Areas with a revenue factor greater than 1 indicate that they require more revenue than contained in the set of activities to pay for their access and excavation, indicating that at baseline economic parameters, they would be unprofitable and thus should not be mined as a part of the mine plan. Areas deemed uneconomic were removed from the schedule.

Table 16-26 summarises the key parameters used in the orphan analysis.

Table 16-26: Key orphan analysis parameters

Parameter	Unit	Value
Development – Capital – Lateral	CA$/m	10,000
Development – Operating – Lateral	CA$/m	8,000
Development – Capital – Vertical	CA$/m	12,000
Mining Costs - Longhole	CA$/t mined	30.5
Backfill Costs	CA$/t mined	9.0
Allocated Milling, G&A Costs	CA$/t mined	14.6
Revenue Factor	CA$/NSR	1.00

16.4.6 Underground Mining Schedule

Activity and Equipment Rates

Underground development rates are inclusive of the time taken to drill, blast, ventilate, muck, and install ground support. Rates are based on contractor development with a bolt-boring jumbo. Ventilation raises and escapeway raises will be excavated by raisebore. Scheduled lateral and vertical development advance rates are summarised in Table 16-27.

Table 16-27: Development activity and equipment rates

Activity	Units	Rate
Lateral development		
Single heading – Primary Decline	m/day	5
Single heading – Capital P1	m/day	4
Single heading – Operating P2[1]	m/day	2
Equipment capacity	m/day	8
Vertical development – Raisebore		
Single heading	m/day	1.8
Equipment capacity	m/day	1.8

Note: A 25% penalty has been applied to slot drive rates.

An excavation rate of 5 m/day is applied only to the main decline access which will have flexibility for independent firing times before reaching the deposit. Production tasks are broken into specific activities, due to the different task rates, equipment required and task durations versus equipment utilization for each task. Some tasks, such as stope blasthole drilling, can fully use equipment capacity until it is completed, while other tasks, such as stope excavation, occupy only a portion of the daily uptime of equipment to complete the task.

Task rates and fixed activity durations for longhole mining are summarised in Table 16-28.

Table 16-28: Longhole production task rates and durations

Activity	Units	Activity rate	Duration
Cable bolting[1]	cbm/d	100	-
Slot Raise[2]	mV/d	5	-
Blasthole Drilling[3]	drm/d	200	-
Stope Excavation – 100 m tram[4]	t/d	950	-
Stope Excavation – 300 m tram	t/d	570	-
Backfill[5]	m^3/d	2,880	-
Backfill Cure	d	-	28

1. cbm = cable bolt metres = 8 x 6m cable bolts at the brow
2. mV = Vertical development metres = Stope height – 5.0 m for drift excavation
3. drm = drill metres = 1 drill metre per 10 t of the stope
4. Total loader productivity = 2,850 tonnes per day
5. Fill volume = diluted mined volume + 100m^3 brow offset allowance

Drilling activities adjacent to backfill can commence as early as 7 days following the completion of backfill, and any excavation activities are delayed the full 28-day cure time.

Lateral Development

There are up to five jumbos proposed for mine development to meet the estimated annual lateral development requirements. The annual lateral development is illustrated in Figure 16-20, while the annual and total lateral development is summarised in Table 16-29.

Figure 16-20: Annual lateral development for KUG

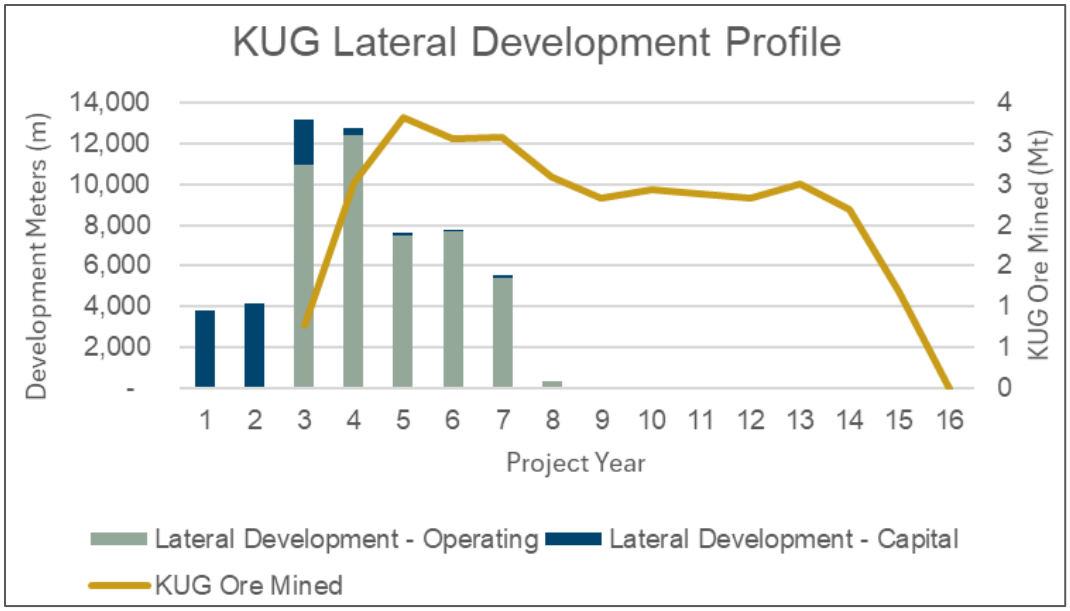


Table 16-29: Total and annual lateral development schedule for KUG

Item	Units	LOM	1	2	3	4	5	6	7	8	9
Lateral development – Capital	m	**10,852**	3,830	4,146	2,201	329	120	64	162	-	-
Lateral development – Operating	m	**44,335**	-	45	10,978	12,415	7,513	7,663	5,383	338	-

Vertical Development

Vertical development is proposed to be completed by raisebore with hole diameters of 1.8 m to 5.0 m. The target advance rate is 1.8 m per day.

The total and annual vertical development is summarised in Table 16-30. Vertical development activities include the excavation of ventilation raises, egress raises, and ore passes.

Table 16-30: Total and Annual Vertical Development Schedule for Kemess Underground

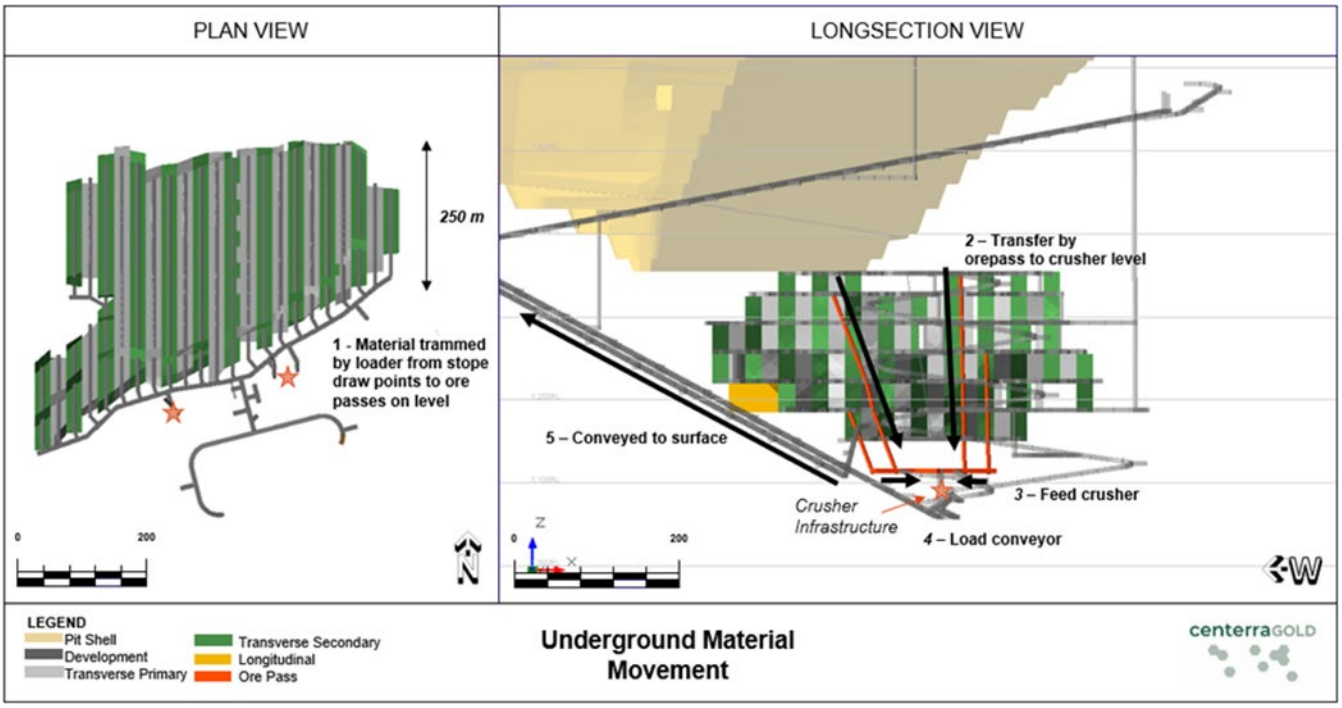
Item	Unit	LOM	1	2	3	4	5	6	7	8	9
Vertical development – Capital	m	**2,096**	133	6	1,172	254	191	232	108	-	-

Material Movement

Waste material haulage for Kemess is proposed to be completed by 21-t capacity loaders and 60–63-t capacity trucks. Waste rock will be loaded into trucks at an on-level stockpile and then hauled to surface for storage in long term waste storage facilities. Mineralisation will be trammed by 21-t capacity loaders and transferred to a central crusher via ore passes located on level. The crushing circuit then loads the underground conveyor belt which transports the material to surface.

The material movement route is illustrated in Figure 16-21.

Figure 16-21: Main haulage pathways at KUG mine

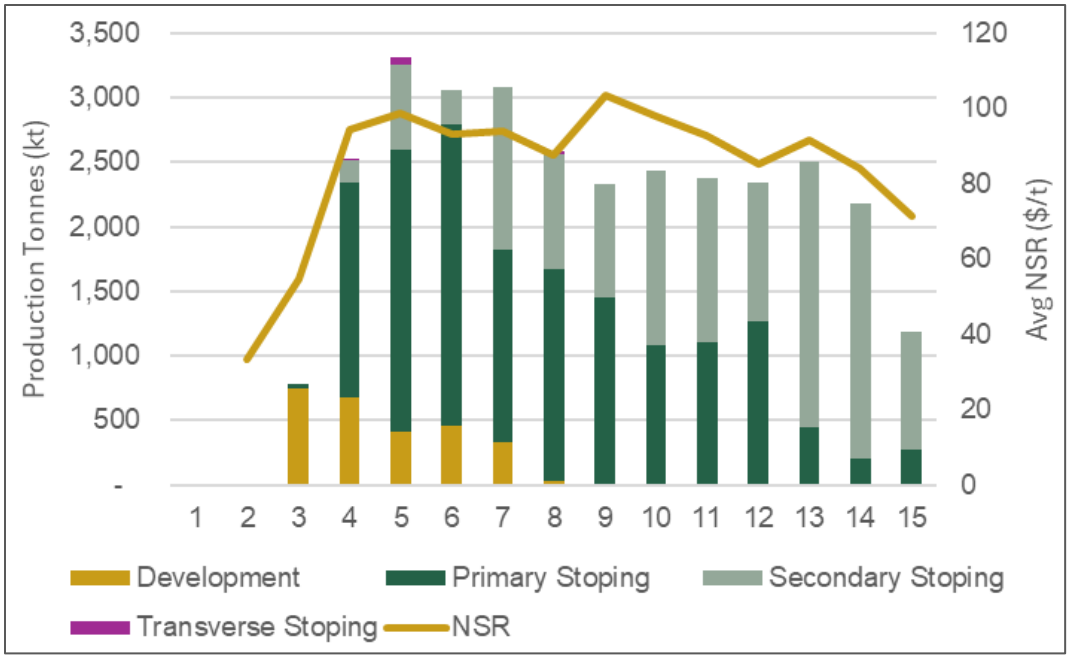


Underground Production Summary

Assuming development can commence in Year 1 for the KUG declines, production could commence in Year 4 and ramp quickly to 1.5 Mtpa due to multiple large stopes being available in Year 3. Production then stabilises between 2.0 Mtpa and 2.5 Mtpa after Year 5 and ramps down after Year 12.

Proposed annual production from underground is illustrated in Figure 16-22.

Figure 16-22: Mined mineralisation for KUG mine



The production from the underground by mineral resource classification is summarised in Table 16-31.

Table 16-31: Total production, by resource classification, for KUG

Classification	Tonnes (Mt)	Grade (% Cu)	Grade (g/t Au)	Grade (g/t Ag)
Total	**30.7**	**0.39**	**0.95**	**2.51**
Measured	-	-	-	-
Indicated	21.5	0.39	0.93	2.58
Inferred	9.2	0.40	0.98	2.36

The annual mined mineralization is summarized in Table 16-32 and Table 16-33 summarizes the tonnes mined by production type.

Table 16-32: Total and annual mine mineralized material schedule for KUG

	Unit	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Total mill feed mined	kt	**30,693**	-	5	776	2,520	3,307	3,062	3,080	2,585	2,335	2,429	2,381	2,336	2,506	2,184	1,186
UG mineralized material to mill	kt	**30,693**	-	5	776	2,520	3,307	3,062	3,080	2,585	2,335	2,429	2,381	2,336	2,506	2,184	1,186
Cu grade to mill – UG	%	**0.39**		0.18	0.27	0.41	0.42	0.40	0.40	0.39	0.42	0.41	0.40	0.38	0.39	0.37	0.32
Au grade to mill – UG	g/t	**0.95**		0.34	0.55	0.96	1.02	0.96	0.97	0.90	1.09	1.01	0.95	0.88	0.97	0.87	0.76
Ag grade to mill – UG	g/t	**2.51**		0.80	1.69	2.26	2.39	2.48	2.47	2.50	2.73	2.73	2.54	2.70	2.54	2.68	2.41

Table 16-33: Total underground mining quantities

Item	Unit	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Development waste	kt	**1,159**	417	447	295	-	-	-	-	-	-	-	-	-	-	-	-
Mineralized development material	kt	**2,654**	-	5	747	672	416	455	336	25	-	-	-	-	-	-	-
Primary stoping	kt	**15,186**	-	-	29	1,676	2,185	2,334	1,486	1,646	1,448	1,086	1,106	1,267	441	205	276
Secondary stoping	kt	**12,764**	-	-	-	170	649	273	1,258	885	887	1,343	1,275	1,069	2,065	1,979	911
Longitudinal	kt	**89**	-	-	-	2	57	-	-	30	-	-	-	-	-	-	-

16.4.7 Backfill

Paste fill is a flexible fill option and can be suited to both bottom-up and top-down mining with minor alteration to the pouring process.

The paste system design is based on limited site-specific test work and therefore relies on a combination of available data, industry practice, and benchmarking against comparable operations.

For the purposes of the PEA, the paste system design considers a single underground distribution system (UDS) operating at a nominal throughput of 200 m³/h. This throughput represents the practical upper limit for a single DN 200 (8”) paste distribution system based on current industry practice and experience; however, actual achievable throughput will be dependent on paste rheology, operating practices, and final system configuration.

Backfill Plant Assumptions

The PEA paste backfill system is based on the following key assumptions and limitations. These assumptions have not been confirmed through project-specific test work and represent key sources of technical uncertainty at the PEA level.

Tailings rheology, settling and filtration performance are assumed to be comparable to benchmarked materials with similar particle size distributions. This presents uncertainty in the preliminarily sized equipment and pumping systems.

Vacuum filtration is assumed to be a suitable technology for Kemess’ specific tailings for paste backfill production. If vacuum filtration is unfavourable and pressure filtration is required, the paste system will become more complex which will affect the design and subsequent costing.

Blended tailings characteristics are assumed to remain reasonably consistent over the life of mine. If tailing characteristics become variable over the LOM, operational impacts may occur. Operational QA/QC is required to monitor tailings characteristic changes.

Binder consumption is determined with assumed strengths typical of stope voids are 20 m x 20 m x 35 m and assuming that 5% of stopes may be mined under backfill (underhand). The estimated binder consumption may change if the stope geometry or the underhand mining factor changes.

The existing tailings pipeline is assumed to be able to manage a lower tailings throughput while tailings are being consumed by the paste system. A review of the tailings pipeline operating envelope is required to minimize the uncertainty.

These assumptions have not been validated through laboratory or pilot-scale testing completed using representative Kemess tailings and process water.

Binder Concentration

Binder concentration assumptions are preliminary and have not been confirmed through site-specific unconfined compressive strength testing. Binder consumption represents a significant component of paste backfill system operating costs. An average binder concentration of approximately 6% by dry mass of solids was estimated for conceptual design purposes, representing a weighted average of plug and fill paste recipes. This value is highly sensitive to tailings mineralogy, particle size distribution, water chemistry, and curing conditions, and therefore remains a key uncertainty at the PEA level. There is no certainty that the assumed average binder concentration will be achieved in practice without completion of site-specific strength testing.

Underground Distribution System

For the purposes of the PEA, the UDS is assumed to be a DN 200 (8”) network of carbon steel pipe with HDPE used for final stope piping. The system will enter the underground at the main access portal and follow the ramp towards the ore body. An underground booster station will be located along this ramp which will receive paste from surface and pump paste to the final deposition point. Each level of the mine, except for the bottom level, will have a network of pipe to access the top of each stope. A series of interlevel boreholes will form the main trunk of the system allowing paste to be delivered to each level. The system will be instrumented for monitoring and control purposes.

Backfill Operations

Utilization of the paste system has been assumed at an industry norm of approximately 60% for the purposes of the PEA and reflects a conceptual operating strategy rather than a detailed production schedule. This utilization rate is intended to account for the intermittent nature of backfilling operations caused by stope availability, underground piping changeovers, operational downtime, and system start-up and flushing.

The throughput and utilization rate is assumed to allow for an overall backfill capacity of approximately 1,000,000 m^3 per annum.

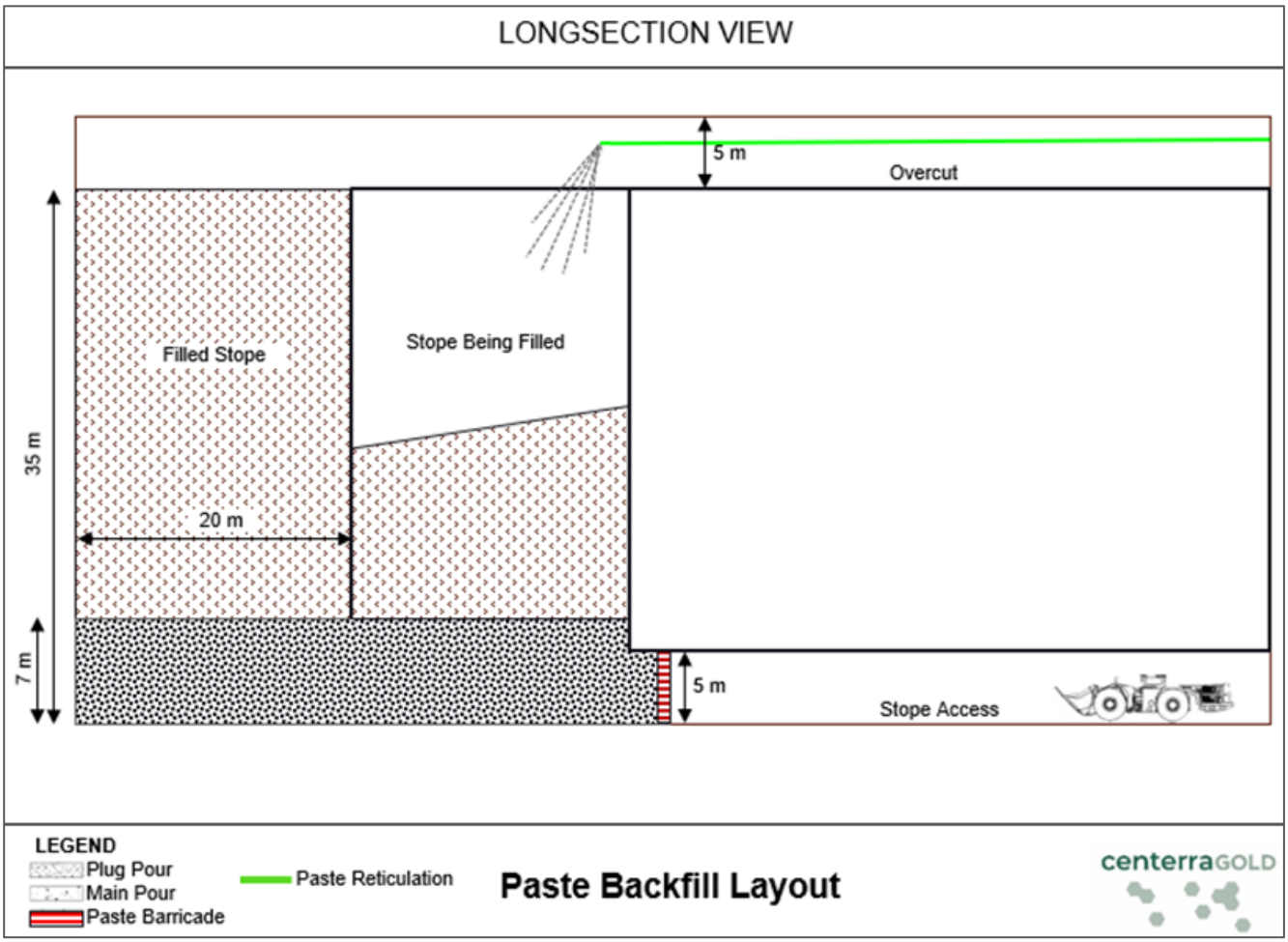
The backfill system proposed for Kemess Mine includes additional thickener tanks and piping infrastructure at the mill, to deliver thickened tailings to the paste fill plant. The paste fill plant and binder storage are planned to be located at the South Portal. An underground booster system is planned to be constructed in the decline to pump the paste fill through the 3.6 km decline to the underground mine distribution system. Further information on the backfill system can be referenced in Item 18-1.

The backfill process considers erecting a barricade at the entrance points of the stope and filling stopes via sufficiently rated piping. After the initial pour (plug) has reached a height above the brow to meet geotechnical recommendations, a 12-hour cure delay is applied that allows for some curing of the paste.

After the delay, the filling of the stope recommences and continues until the void is sufficiently filled. Following a 28-day cure period, drilling activities in adjacent stopes can be scheduled.

A schematic of the paste fill pour and infrastructure is illustrated in Figure 16-23.

Figure 16-23: Schematic of paste backfill activity and associated infrastructure



For costing purposes, the plug is to contain 4.0% binder, and the main body pour to contain 2.5–5.0% binder. The cement content and cure durations are summarised in Table 16-34.

Table 16-34: Backfill binder quantities and curing durations

Activity	Unit	Longhole
Plug Pour	% Binder	4.0
Plug Cure	days	0.5
Main Body Pour	% Binder	2.5–5.0
Body Cure – Production	days	28.0

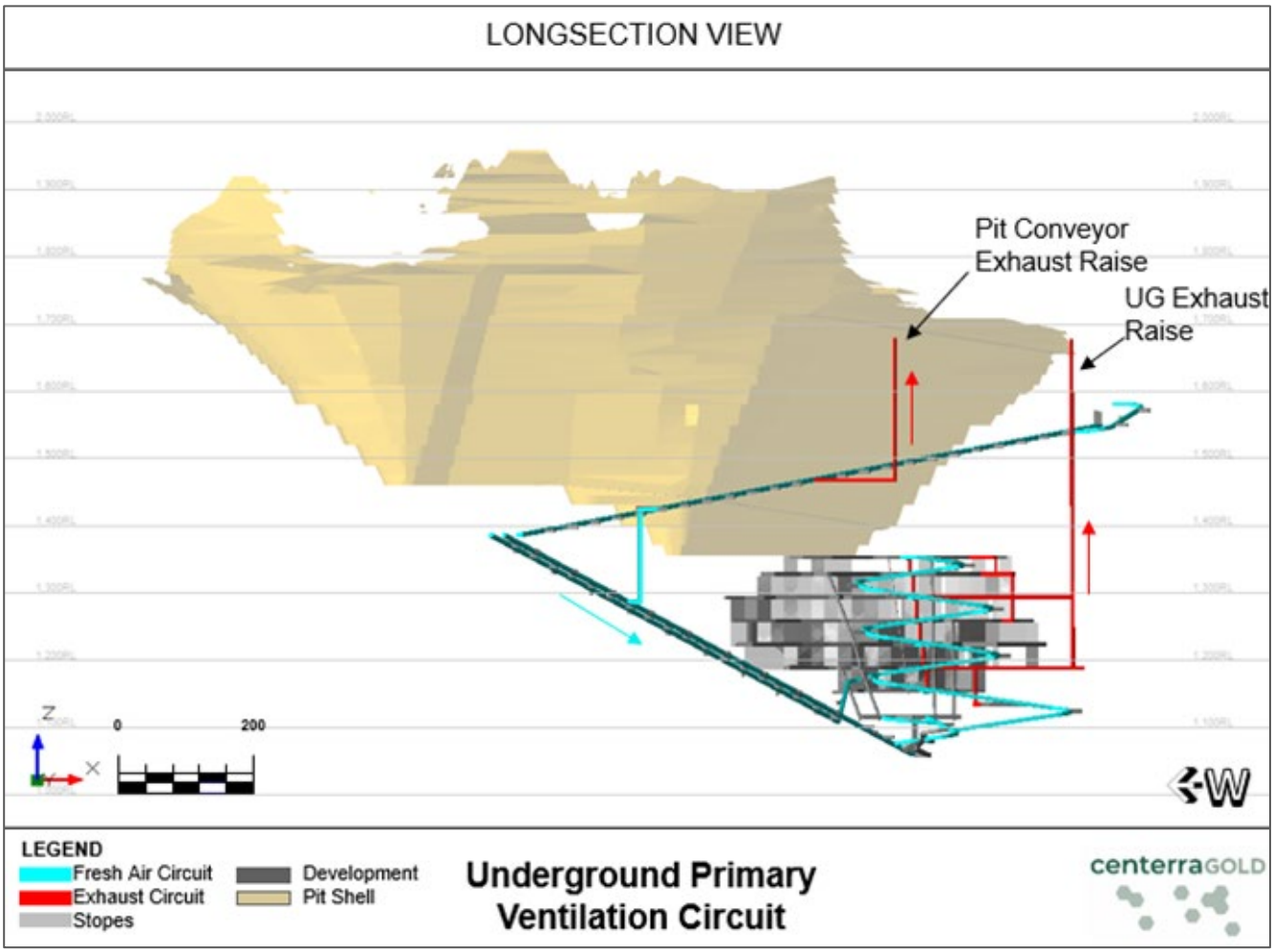
All paste activities are completed at a capacity rate of 2,880 m^3 per day, which includes utilisation, downtime and availability, but excludes cure durations. The paste plant size was specified with one module which limits the productivity of the underground mine to 2.5 Mtpa.

16.4.8 Underground Infrastructure and Services

Primary Ventilation

A preliminary evaluation of the primary ventilation circuit was completed in Ventsim software. Two declines facilitate the ventilation circuit during the development and construction phase of the project. During the production phase of the project, the declines will act as fresh air intake and raises between sublevels will exhaust to a surface raise connection that daylights to the east of the planned open pit. The on-level exhaust raises are designed to allow for rapid clearance of blast fumes. The primary ventilation circuit is illustrated in Figure 16-24. The pit conveyor design employs a ventilation raise to the underground conveyor decline to facilitate ventilation during development.

Figure 16-24: Ventilation circuit at KUG mine



The underground ventilation system design is driven by the regulated threshold air velocity and the regulated air volume required to disperse and exhaust the exhaust of internal combustion diesel engines. Table 16-35 provides a list of mobile mining equipment and the calculated total air volume to ventilate the mine.

Table 16-35: Ventilation demand estimate for the KUG mine

Equipment/Unit	Quantity (#)	Engine power (kW)	Utilization	Demand[1] (m^3/s)
Truck[2]	1	565	100%	33.9
Truck Loader	1	352	100%	21.1
Development Loader	2	352	100%	42.2
Stope Loader	6	352	100%	126.7
Jumbo Drill	3	122	25%	5.5
Bolter	3	122	25%	5.5
Cable bolter	1	119	25%	1.8
Production Drill	6	120	25%	10.8
Raisebore	2	122	25%	3.7
Diamond Drill	2	122	25%	3.7
Charge Wagon	3	120	25%	5.4
Spraymec	1	82	25%	1.2
Agitator	1	170	25%	2.6
IT – Service Crew	1	111	25%	1.7
IT – Paste Crew	2	111	25%	3.3
Fuel Truck	1	250	25%	3.8
Water Truck	1	250	25%	3.8
Grader	1	123	100%	7.4
Light Vehicles	24	110	25%	39.6
Subtotal				**323.5**
Workshop (5%)				16.2
Contingency (10%)				32.4
Mine Leakage (10%)				32.4
Total				**404.4**

1. Total airflow requirements based on regulation code of $0.06 m^3/s$ per engine kW
2. Additional trucks required during development phase. Only production trucks shown.

Auxiliary Ventilation

When headings are located outside of the primary ventilation circuit, auxiliary fans are required to deliver fresh air into the working headings. Auxiliary ventilation is provided via 1,220 mm flexible ducting fed from 110 kW fans located in fresh air. Table 16-36 estimates the typical air volume needed for levels during mine development.

Table 16-36: Level ventilation demand estimate for KUG mine

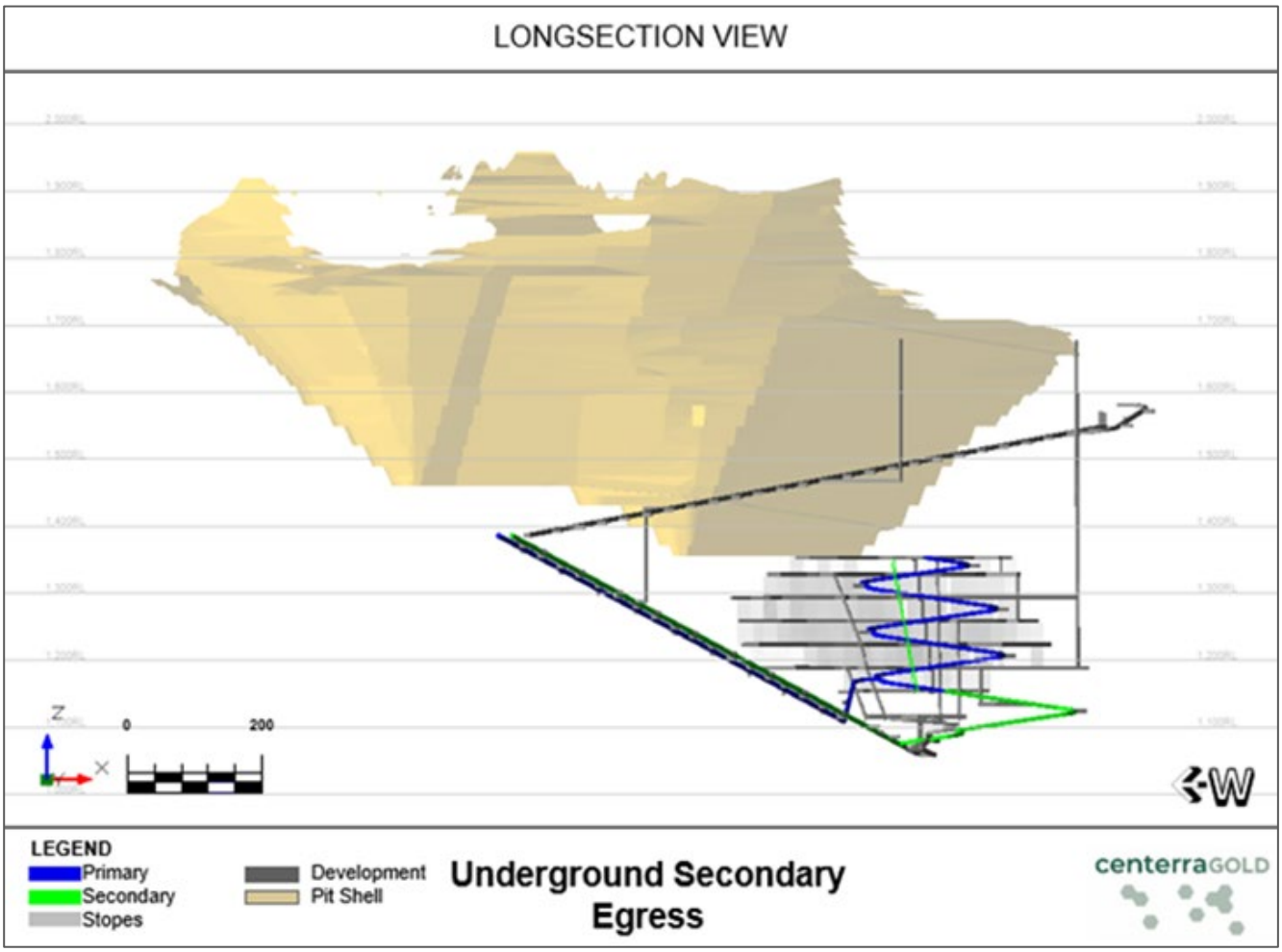
Equipment/Unit	Model	Quantity (#)	Engine power (kW)	Utilization	Demand[1] (m^3/s)
Trucks	TH545	1	450	100%	27.0
Loaders	LH517	1	310	100%	18.6
Subtotal					**45.6**
Contingency[2]					4.6
Total					**50.2**

1. Total airflow requirements based on $0.06 m^3/s$ per engine kW
2. Contingency of 10% to account for ducting condition

Secondary Means of Egress

A second means of mine egress, required by regulations, will be via either the twin declines or via connecting escapeways between levels as illustrated in Figure 16-25. All level designs contain a secondary means of egress between each production level to prevent entrapment. Escapeways are located separately from the exhaust ventilation circuit to reduce the introduction of smoke and other contaminants to the egress circuit.

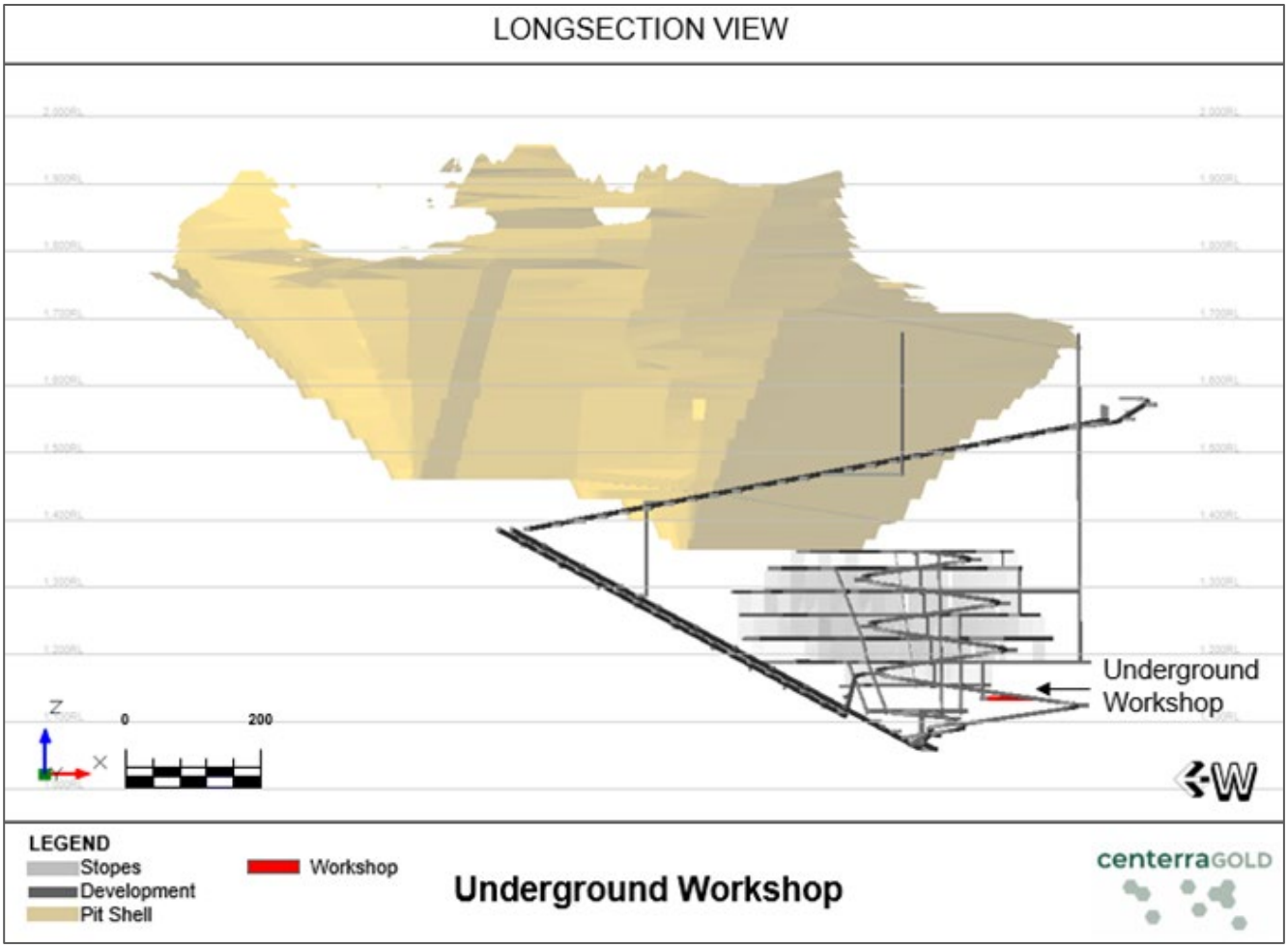
Figure 16-25: Secondary egress for Kemess Mine



Underground Workshop

Excavations for an underground workshop have been included in the design and schedule. The location of the workshop is illustrated in Figure 16-26. Detailed design has not been completed at this stage of study, and capital allocation has been included in the mine cost modeling to support equipping the workshop.

Figure 16-26: Underground workshop at KUG mine



Crushing and Conveying

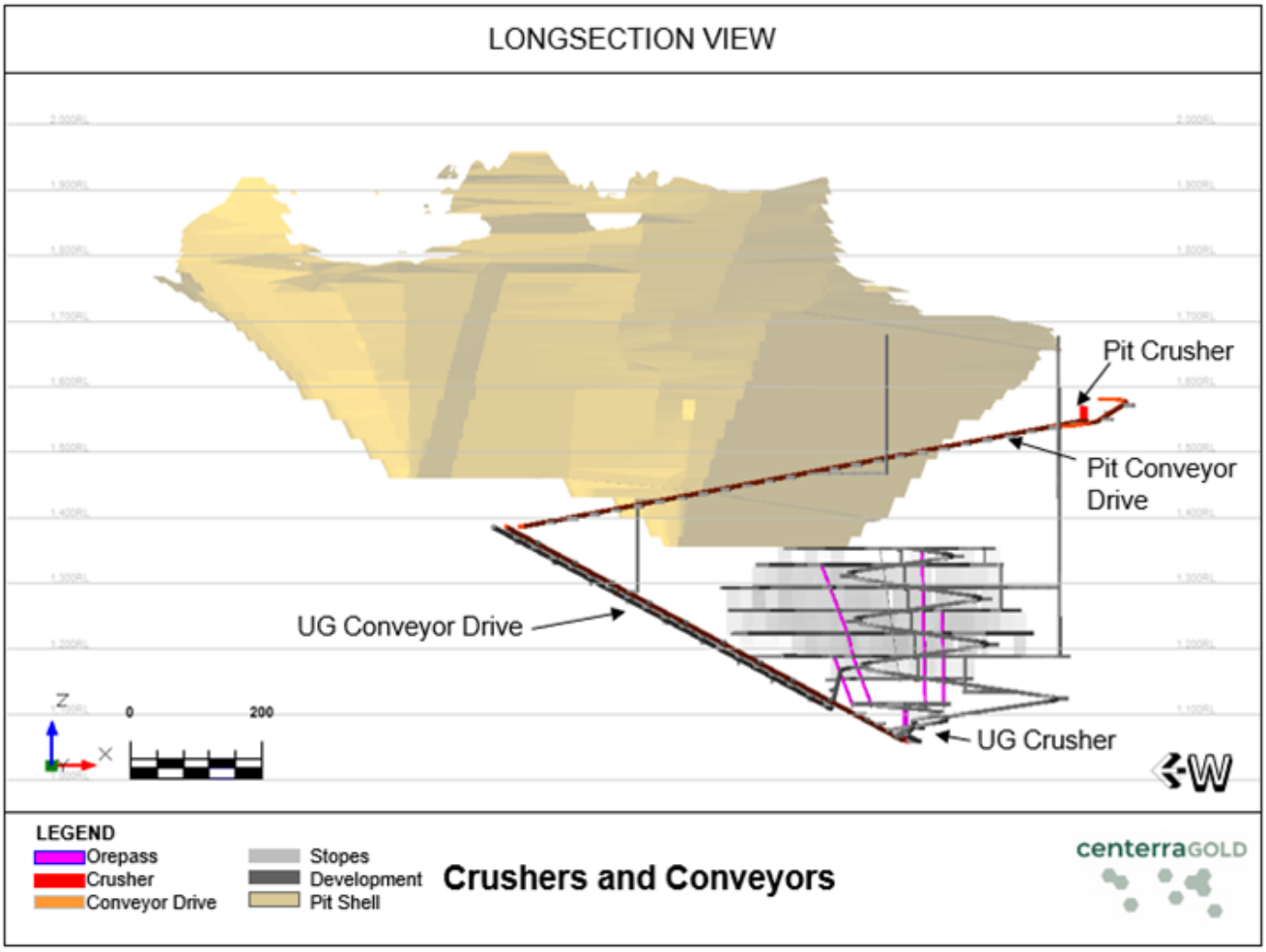
A conveyor was designed to be the primary material movement method for KUG. A second conveyor, including underground development access, was included in the design to transfer material from the open pit to the processing facilities, avoiding a long overland conveyor system around the mountain.

The conveyor supporting KUG is 3.2 km long and designed at a 10% gradient. The conveyor is designed to be fed by crushing infrastructure located at the bottom of the conveyor. The design includes two storage bins with 875 m^3 capacity to allow for continuous feed to crusher infrastructure between shifts.

The open pit conveyor is 3.9 km long and designed at -4% gradient. The conveyor has crushing infrastructure located at the north end of the conveyor, shown by Figure 16-27. The negative gradient of the conveyor may potentially generate power; however, the benefits have not been included in this preliminary assessment.

Detailed designs for crusher and conveyor infrastructure including mass excavations for crusher rooms, sizing and loading, have not been completed as part of this study. Preliminary cost estimates have been included in Item 21-2 of the report.

Figure 16-27: Crushers and conveyors at KUG mine



Water Management

For costing purposes, helical-rotor style pumps are proposed to pump mine water at a rate of 40 L/s from underground via an 8” Sched 40 pipe. Detailed water management designs have not been completed at this stage of study. Total vertical pumping height is approximately 350 m pumping stations proposed at every 240 vertical metres. Mine water is collected by on-level sumps and directed to the lower main pumping station.

Compressed Air

Detailed compressed air designs have not been completed at this stage of study. Allocations for required compressed air infrastructure and distribution have been included in the cost estimate.

Electrical power

Detailed electrical designs have not been completed at this stage of study. Preliminary estimations of power and diesel consumption have been completed based on first principles. Underground mine power is estimated at an average of 23 kWh/t and 0.89 L/t for diesel consumption.

Two surface ventilation fans demand approximately 2.5 MW of power (total, installed), conveying and crushing (2.5 MW), and underground ventilation fans (0.8 – 2.8 MW). Energy consumption by auxiliary fans reduces as the mine transitions into the production phase. The estimated power and diesel consumption for KUG are summarised in Table 16-37.

Table 16-37: Kemess power estimate (mining activities only)

Year	Energy consumption (MWh)	Diesel (kL)
Year -3	9,443	1,167
-2	23,941	1,553
-1	32,441	2,245
Year 1	51,542	2,504
1	64,061	2,830
2	63,242	2,776
2	61,603	2,650
3	56,689	1,929
4	52,594	1,543
5	52,594	1,543
6	51,775	1,343
7	51,775	1,343
8	52,594	1,543
9	50,956	1,217
Total / Max	**675,248**	**26,188**
Average	22.3 kWh /t	0.89 L/t

16.4.9 Underground Mine Personnel

The underground workforce relates to direct labour for underground operations and excludes the surface operations of processing, site administration including management and support, health and safety, environmental, surface logistics, and security. See Item 16.6.

16.5 MINING SCHEDULE

The combined open pit and underground production schedule is included in Figure 16-28 and Table 16-38. The mill feed blend prioritizes underground production due to its relatively higher head grade. The open pit ore delivery schedule is adjusted to balance the mill feed rate. On average, underground material will constitute 14% of the mill feed.

Figure 16-28: Annual combined open pit and underground production and metal grades

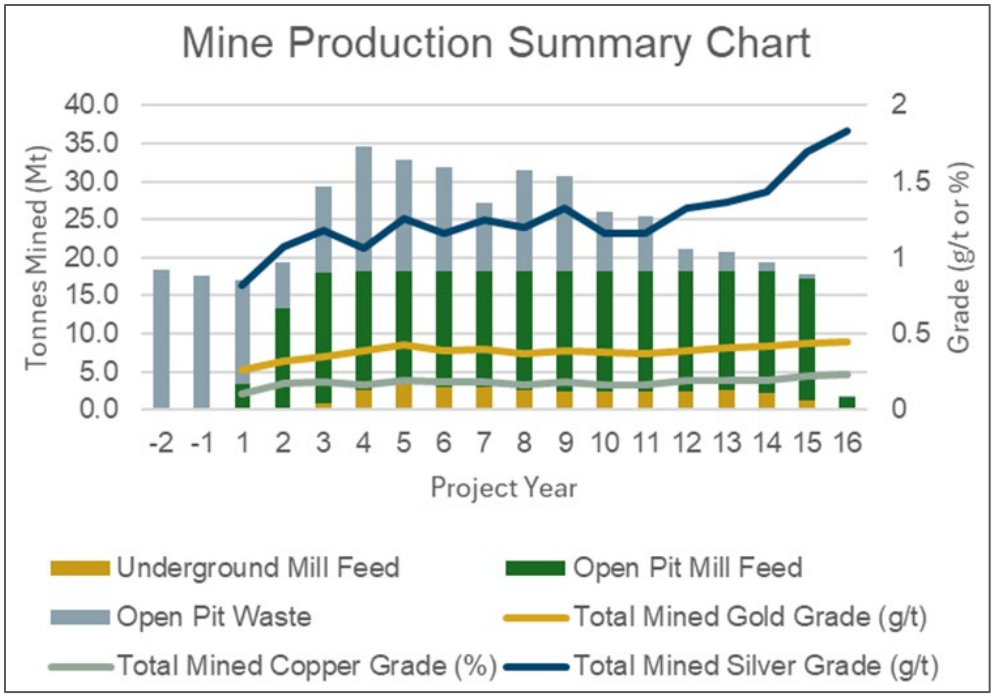
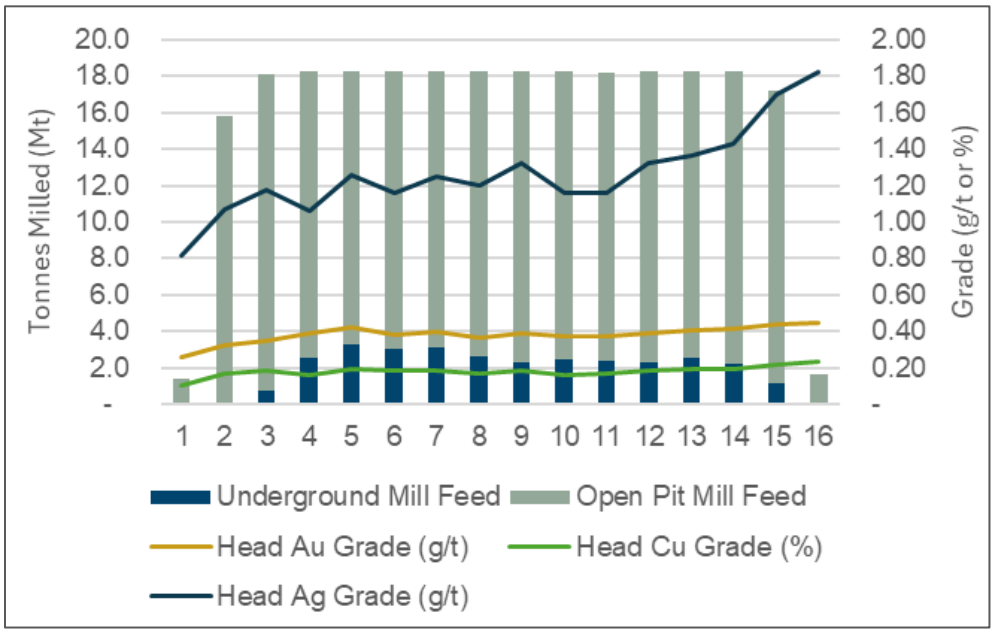


Table 16-38: LOM open pit and underground ore delivery to Process Plant

Item	Unit	LOM	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
OP mineralized material to mill – non broken zone	**Mt**	**146.8**	**0.0**	**0.0**	**3.3**	**2.4**	**3.9**	**5.6**	**9.5**	**12.2**	**15.2**	**13.9**	**15.5**	**15.9**	**15.7**	**16.1**	**16.1**	**1.7**
Cu grade to mill – non broken zone	%	**0.15**	0.08	0.08	0.18	0.11	0.12	0.12	0.14	0.13	0.15	0.13	0.13	0.16	0.16	0.17	0.21	0.24
Au grade to mill – non broken zone	g/t	**0.32**	0.19	0.18	0.38	0.30	0.28	0.27	0.29	0.29	0.29	0.28	0.28	0.32	0.32	0.35	0.41	0.44
Ag grade to mill – non broken zone	g/t	**1.16**	0.67	0.79	1.38	0.88	1.04	0.91	1.09	1.06	1.15	0.97	0.96	1.12	1.18	1.26	1.65	1.83
OP mineralized material to mill – broken zone	**Mt**	**77.4**	**1.4**	**15.8**	**14.0**	**13.3**	**11.1**	**9.6**	**5.7**	**3.4**	**0.7**	**1.9**	**0.4**	**0.0**	**0.0**	**-**	**-**	**-**
Cu grade to mill – broken zone	%	**0.15**	0.10	0.17	0.18	0.12	0.15	0.15	0.14	0.11	0.10	0.11	0.10	0.08	0.10	-	-	-
Au grade to mill – broken zone	g/t	**0.30**	0.26	0.32	0.33	0.30	0.29	0.27	0.26	0.25	0.24	0.26	0.24	0.19	0.18	-	-	-
Ag grade to mill – broken zone	g/t	**0.95**	0.82	1.07	1.10	0.87	0.99	0.89	0.85	0.72	0.55	0.52	0.61	0.66	0.48	-	-	-
OP mineralized material to mill – subtotal	**Mt**	**224.2**	**1.4**	**15.8**	**17.3**	**15.7**	**14.9**	**15.2**	**15.2**	**15.7**	**15.9**	**15.8**	**15.8**	**15.9**	**15.7**	**16.1**	**16.1**	**1.7**
Cu grade to mill – OP	%	0.15	0.10	0.17	0.18	0.12	0.14	0.14	0.14	0.13	0.15	0.13	0.13	0.16	0.16	0.17	0.21	0.24
Au grade to mill – OP	g/t	0.31	0.26	0.32	0.34	0.30	0.29	0.27	0.28	0.28	0.29	0.28	0.28	0.32	0.32	0.35	0.41	0.44
Ag grade to mill – OP	g/t	1.09	0.82	1.07	1.15	0.87	1.00	0.90	1.00	0.99	1.12	0.92	0.95	1.12	1.18	1.26	1.65	1.83
UG mineralized material to mill – subtotal	**Mt**	**30.7**	**-**	**0.0**	**0.8**	**2.5**	**3.3**	**3.1**	**3.1**	**2.6**	**2.3**	**2.4**	**2.4**	**2.3**	**2.5**	**2.2**	**1.2**	**-**
Cu grade to mill – UG	%	**0.39**	-	0.18	0.27	0.41	0.42	0.40	0.40	0.39	0.42	0.41	0.40	0.38	0.39	0.37	0.32	-
Au grade to mill – UG	g/t	**0.95**	-	0.34	0.55	0.96	1.02	0.96	0.97	0.90	1.09	1.01	0.95	0.88	0.97	0.87	0.76	-
Ag grade to mill – UG	g/t	**2.51**	-	0.80	1.69	2.26	2.39	2.48	2.47	2.50	2.73	2.73	2.54	2.70	2.54	2.68	2.41	-
Total mineralized material to mill	**Mt**	**254.9**	**1.4**	**15.8**	**18.1**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**18.2**	**17.2**	**1.7**
Cu grade to mill	**%**	**0.18**	**0.10**	**0.17**	**0.18**	**0.16**	**0.19**	**0.18**	**0.18**	**0.17**	**0.18**	**0.16**	**0.17**	**0.19**	**0.19**	**0.20**	**0.22**	**0.24**
Au grade to mill	**g/t**	**0.39**	**0.26**	**0.32**	**0.35**	**0.39**	**0.42**	**0.39**	**0.40**	**0.37**	**0.39**	**0.38**	**0.37**	**0.39**	**0.41**	**0.41**	**0.44**	**0.44**
Ag grade to mill	**g/t**	**1.26**	**0.82**	**1.07**	**1.18**	**1.06**	**1.25**	**1.16**	**1.25**	**1.20**	**1.32**	**1.16**	**1.16**	**1.33**	**1.36**	**1.43**	**1.70**	**1.83**

The schedule of mine production is then used to feed the processing plant, drawing directly from the open pit mine, then adding the underground and drawing from the stockpile as described above. A schedule of mineralized material to the process plant is illustrated by Figure 16-29.

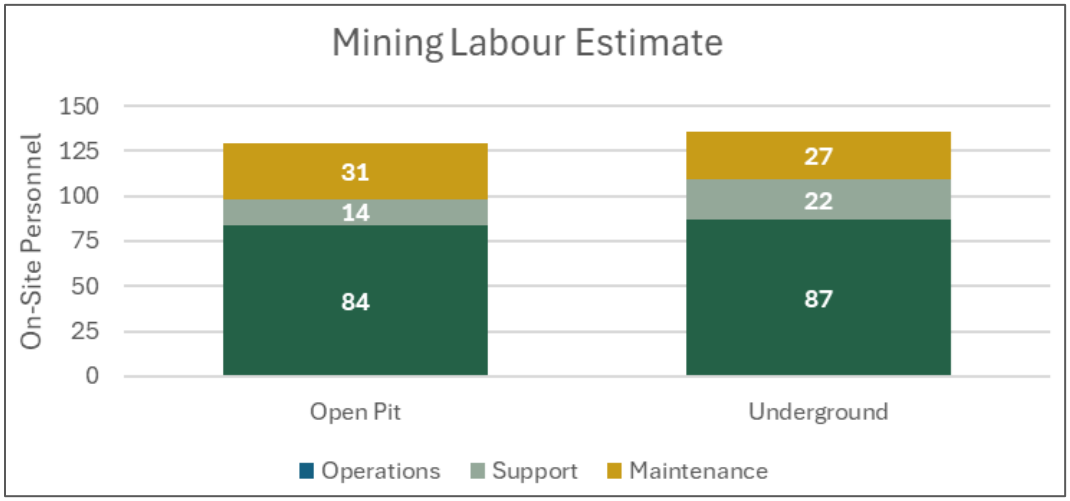
Figure 16-29: Process feed summary



16.6 MINING PERSONNEL

The proposed mining operation is planned for round-the-clock operation with workers rotating on a two-week on/two-week off roster. Senior roles and operations support roles will be on a 4-days on/3-days off roster. The estimated steady-state labour complement is 265 on-site personnel. The staffing split between open pit and underground is shown by Figure 16-30.

Figure 16-30: On-site personnel



16.7 OPEN PIT AND UNDERGROUND INTERACTION

Underground production from stoping will advance from deeper elevations to shallower elevations, stopping at 1,355 masl. Open pit mining will reach 1,355 masl after underground mining reaches that elevation from below. Stopes will be backfilled well in advance of open pit benching reaching the elevation.

17 RECOVERY METHODS

17.1 SUMMARY

The plant operated from 1999 to 2011 and has been on a care and maintenance program since. As such, there is significant infrastructure already in place, and the existing equipment will be refurbished where possible and supplemented with new replacement equipment. The plant will process material at a nominal rate of 18.250 Mtpa for 16 years with an average head grade of 0.15% Cu and 0.37 g/t Au. The plant is designed to operate 2 shifts per day, 365 days per year with an overall plant availability of 92%.

The process plant feed will be supplied by the Main Zone and the Underground deposits. The process plant will produce gold rich copper concentrate to be sold to smelters.

The processing plant will consist of the following:

- Single stage crushing circuit (gyratory), fed from the open pit mine
- Single stage crushing circuit (jaw), fed from the underground mine
- Underground and overland material handling conveying systems
- Coarse material stockpile with reclaim system
- Two primary grinding circuits comprised of a SAG mill and a ball mill in closed circuit with hydrocyclones
- Rougher flotation
- Conventional concentrate regrind on rougher concentrate
- Secondary, high-intensity regrinding on 1st cleaner concentrate
- Cleaner-scavenger flotation
- Conventional CIP and ADR leach plant to process cleaner-scavenger tailings
- Three stages of cleaning, including high-intensity flotation;
- Concentrate thickening, filtration and storage;
- Process water systems
- Plant air and utility systems
- Reagent systems
- Bulk concentrate load-out by front-end loader filling concentrate transportation
- Final tailings pumping to the tailings storage facility (TSF).

17.2 PLANT DESIGN

17.2.1 Overview

Mineralized material from the Main Zone will be crushed to a top size of 150mm and conveyed to a transfer tower at the underground portal. ROM material from the underground mine will also be crushed to a top size of 150mm and conveyed to the underground portal. Mineralized material from the Main Zone and underground will be combined at the portal and the comingled material will be conveyed to a coarse material stockpile.

Two parallel milling lines will feed coarse mineralized material from the stockpile to the SAG mill feed conveyors. The A milling line will be fitted with new equipment while the B milling line will use existing, refurbished equipment. Three apron feeders per milling line will transfer material to the SAG mill feed conveyor, which will feed the SAG mill. The target material size introduced into the SAG and ball mill circuits will be 80% passing (P80) 60 mm, or finer, product. Oversize from the SAG discharge screens will be recycled back to the SAG mill by refurbished high-angle conveyors for reprocessing. The design for the final feed to flotation from the ball mill circuit will be designed to achieve a product size of P80< 42 µm (0.04 mm).

The refurbished rougher flotation circuit will produce a rougher concentrate that will be directed to the first stage regrind circuit. The rougher concentrate will be classified in a regrind cyclone cluster, with coarse material sent to the refurbished regrind ball mill which will target a regrind product of approximately 40 µm and a portion treated by a refurbished gravity concentrator. The refurbished regrind cyclone overflow will feed the first cleaner flotation (refurbished cells) followed by cleaner scavenger flotation (refurbished cells).

Concentrate from the first cleaner and cleaner-scavenger stages will be further reduced in size in a new high-intensity grinding mill (HIGmill™) equipped with a Cyclopac, targeting a product P80 of <20 µm. The HIGmill product will be transferred to the secondary cleaner flotation, which will consist of a new Jameson Cell. Secondary cleaner concentrate will advance to the third cleaner flotation column (refurbished) and then to the refurbished scavenger flotation column for final upgrading.

Tailings from the cleaner-scavengers will be sent to a conventional CIP leaching plant. The ADR portion of the plant will produce gold as doré and the tailings from the leach plant will discharge into the old Kemess south pit for the LOM.

The resulting final flotation concentrate is expected to target approximately 23% Cu, with these targets adjustable depending on metal prices and smelter terms. Gold recovered in the gravity concentrator will be directly fed to the final concentrate thickener, bypassing the cleaner circuit.

Final flotation concentrate will be thickened, dewatered, and pressure-filtered to a target moisture content of 8%, stockpiled, and then trucked to the rail loadout facility in Mackenzie, BC. The concentrate will then be railed to North Vancouver, where it will be loaded onto ships and transported to purchasers.

Rougher tailings and the concentrate clarifier overflow will be combined into a single tailings stream and pumped to the TSF. No separate leach tailings will be generated, as the new CIL–ADR circuit processes cleaner scavenger concentrate. Tailings deposition will be split between the two facilities, the old Kemess south pit and TSF, and will vary over the life of mine.

17.2.2 Throughput Reinstatement

The Kemess South mill was capable of processing approximately 50,000 tonnes of material per day. Following the shut down of Kemess South, one of the two parallel grinding lines, line ‘A’, consisting of a SAG mill, ball mill and associated ancillary equipment was removed and the mills were sold.

Upon restart, the mine throughput will be reinstated to 50,000 tonnes of material per day. To reinstate the mill to a 50,000 tonnes per day throughput, additional grinding capacity will be added to the circuit through the installation of a new SAG mill and new ball mill in Line A. These will be similarly sized to the existing mills.

17.2.3 Process Design Criteria

The process design criteria used as a basis for the current flowsheet were developed using feed characteristics and information from the mine development, metallurgical test work, current installed equipment sizing, and vendor information, summarized on Table 17-1.

Table 17-1: Kemess process design criteria

Parameter	Units	Value
Annual Process Plant Throughput, Design	Mt/a	21.9
Daily Process Plant Throughput, Design	t/d	60,000
Daily Process Plant Throughput, Nominal	t/d	50,000
Copper Head Grade	%	0.15
Gold Head Grade	g/t	0.37
Life Of Mine	y	16
Operating Availability, Crushing	%	70
Operating Availability, Overall Plant	%	92
Copper Recovery, Design	%	89.1
Gold Recovery, Design	%	61.0
JK SMC Axb, Design	-	40.8
SMC Mia Parameter	kWh/t	19.4
SMC Mib Parameter	kWh/t	20.5
SMC Mic Parameter	kWh/t	7.4
Bond Abrasion Index, Design	-	0.06

Parameter	Units	Value
Specific Gravity	-	2.8
SAG Mill Circulating Load, Design	%	29
Ball Mill Circulating Load, Design	%	380
Grinding Feed Size, F80	mm	109
Grinding Product Size, P80	µm	189
Ball Mill Cyclone Overflow Density	% w/w solids	36
Rougher Flotation Mass Pull	%	10
Primary Regrind Product Size, P80	µm	42
Secondary Regrind Product Size, P80	µm	20
Final Concentrate Mass Pull (Overall)	%	0.58
Concentrate Thickening Rate, Design	$t/m^2/h$	0.1
Concentrate Specific Filtration Rate	$kg/m^2/h$	610

17.3 PROCESS PLANT DESCRIPTION

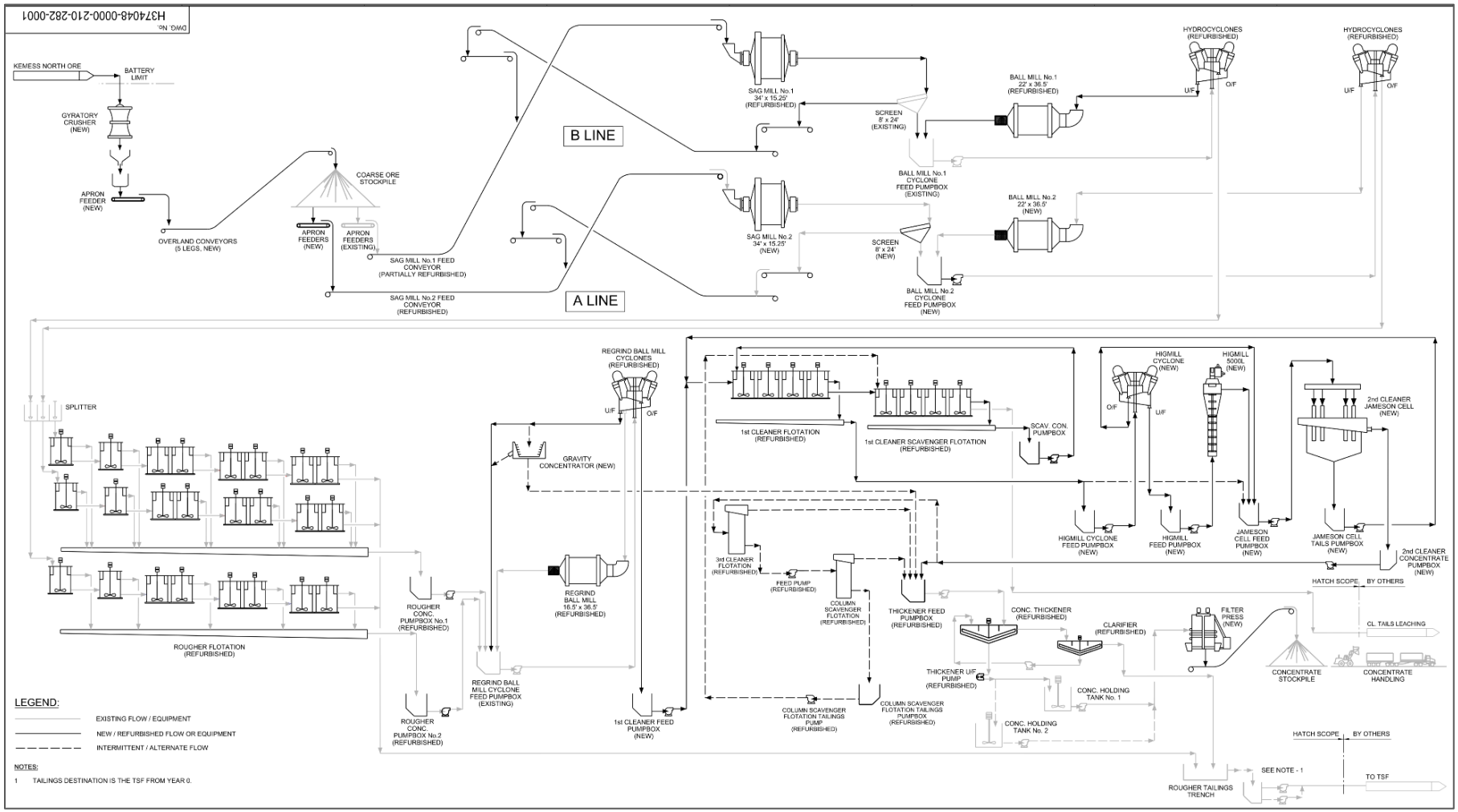
The process design has the following major components:

- Primary crushing
- One coarse material stockpile and two conveyors, one for each comminution line
- Two lines of SAG and ball millgrinding with cyclone classification
- Three parallel rougher flotation lines with a common concentrate cyclone classification and regrind ball mill
- Three stages of cleaner flotation
- Concentrate thickening, filtration and stockpiling
- Final tailings thickening and pumping to the tailings storage facilities (TSF).

17.3.1 Process Flowsheet

The overall process flow diagram is shown by Figure 17-1.

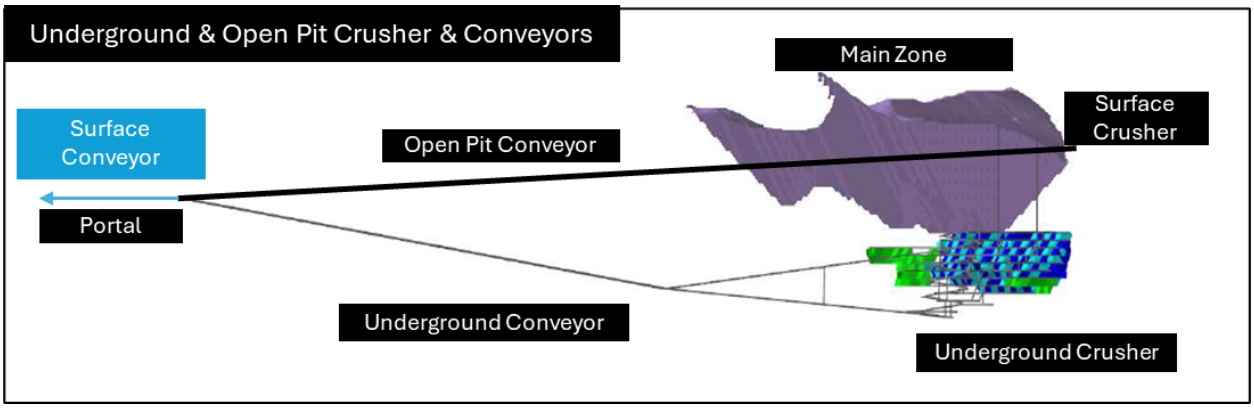
Figure 17-1: Kemess overall process flow diagram



17.3.2 Crushing

The project will need two crushers and the associated material handling to transport ore from Main Zone and the underground to the existing mill. Figure 17-2 shows the concept, where the surface crusher will connect to the south infrastructure via a tunnel that houses a conveyor that daylights at the underground portal. The underground crusher and conveyor will merge with the open pit conveyor at a transfer point at the portal. From there a surface conveyor will transport the comingled ore to the mill.

Figure 17-2: Underground and open pit crusher and conveyor routing



Open pit Main Zone mineralized material will be delivered to a gyratory crusher that is designed to process 50,000 tpd at 70% availability and produce a P80 product size of 150 mm. The underground jaw crusher is designed to process 8,000 tpd at 70% availability, delivering a product size of 150 mm.

Crushed mineralized material from both circuits will be transported via the underground and overland conveyor systems to the coarse material stockpile adjacent to the process plant, which supplies mineralized material through a reclaim system to the two downstream comminution trains and provides surge capacity.

Major equipment in this area includes the following:

- Primary gyratory crusher – Surface (new)
- Primary jaw crusher – Underground (new)

17.3.3 Overland Conveyor

Overland conveying will be utilized to convey mineralized material from the primary gyratory crusher and the primary jaw crusher to the process plant. The conveyor routing will be optimized to account for topography and will travel through two tunnels. The downhill segments of the conveyor system will be equipped with reversing drives capable of generating electricity to minimize electrical power usage. Five legs of overland conveyors with variable frequency drives will be utilized to convey material from the

crushers to the coarse material stockpile near the process plant. All overland conveying systems will be new equipment.

17.3.4 Primary Stockpile

Ore will be transferred from the overland conveyor to the coarse ore stockpile adjacent to the process plant using the existing stacker conveyor. The coarse ore stockpile has a live storage capacity of 74,200 t. Ore will be reclaimed from the stockpile through two lines which each have three existing 1.83 m x 6.1 m (6 ft x 20 ft) apron feeders. The ore is delivered to the two grinding lines via two 1.52 m (60-inch) wide SAG mill feed conveyors.

17.3.5 Grinding Circuit

The grinding circuit will consist of two parallel single-train SAG and ball mills (Line A and Line B). One train is existing and will be refurbished, and one new train will be installed to replace Line A which was removed after the Kemess Mine ceased operations. New higher power motors will be installed in the existing line and the new line, increasing the grinding power in the plant to account for the increased ore hardness.

Line B is comprised of a 10.36 m diameter x 4.65 m (34 ft x 15.25 ft) SAG mill driven by two new 5,250 kW (7,040 hp) motors. The SAG mill is variable speed and will operate in series with a closed-circuit 6.7 m diameter x 11.1 m (22 ft x 36.5 ft) ball mill, also driven by new twin 5,250 kW (7,040 hp) motors.

The Line A second train will be comprised of new mills that will match the dimensions of the original mills but will have larger installed power. A new 10.36 m diameter x 4.65 m (34 ft x 15.25 ft) SAG mill driven by two 5,250 kW (7,040 hp) motors will be installed. The SAG mill will be variable speed and will operate in series with a closed-circuit 6.71 m diameter x 11.13 m (22 ft x 36.5 ft) ball mill, driven by twin 5,250 kW (7,040 hp) motors.

Coarse material or pebbles discharging from the SAG mill (screen oversize) will be returned to the mill without being crushed. Slurry discharging from the SAG mill (screen undersize) and ball mill will be collected in the cyclone feed pump box and pumped to the ball mill cyclopac for closed circuit ball mill grinding.

Major equipment in this area includes the following:

- Line B
  - SAG mill 34’ x 15.25’ (new)
  - Screen 8’ x 24’ (new)
  - Ball mill 22’ x 36.5’ (new)

- Hydrocyclone (refurbished)

- Line A
  - SAG mill 34’ x 15.25’ (refurbished)
  - Screen 8’ x 24’ (refurbished)
  - Ball mill 22’ x 36.5’ (refurbished)
  - Hydrocyclone (refurbished)

17.3.6 Flotation and Fine Grinding

Grinding cyclone overflow from Line A and Line B will be transferred by gravity to the existing flotation feed distribution box, which will divide the flow between three parallel rougher flotation lines. Each rougher bank consists of eight 130 m³ Wemco® rougher cells and a feed distribution tank, enabling independent operation of each bank. During operations of Kemess South the first four cells in each bank were converted to Outokumpu, forced air cells. Planned refurbishment works will maintain this configuration.

The rougher concentrate will feed the regrind circuit pumpbox, where it will be directed to the regrind cyclone cluster. The cyclone underflow (coarse fraction) will report to the existing 16.5 ft × 36.5 ft regrind ball mill (1,100 kW), which will target a regrind product of approximately 40 µm. A gravity concentrator will treat a portion of the circulating load in the underflow; returning the discharge to the pumpbox. The cyclone overflow will proceed to the first cleaner flotation, which will be the re-purposed fourth rougher line consisting of eight 130 m³ tank cells. The first four cells will be the first cleaners and the last four cells will be the cleaner scavengers. In total, the cleaner and cleaner scavengers will comprise a total of eight 130 m³ cells.

First cleaner concentrate will be processed in a high intensity grinding mill (HIGmill™) with an installed power of 1,100 kW, equipped with a cyclopac for classification. The HIGmill™ will operate to achieve a fine product near 20 µm, with its product transferred to the secondary cleaner stage, which consists of one Jameson Cell.

Tailings from the Jameson Cell will be returned to the first cleaner for recycle, while the concentrate will advance to the third cleaner column followed by column scavenger flotation. The column flotation concentrate will then be pumped to the dewatering system for final processing, while tailings will be sent back to the first cleaner stage.

Tails from the first cleaner scavengers will exit the plant and proceed to the leach plant for further processing.

Major equipment in this area includes the following:

- Three Wemco® rougher flotation banks 8 x 130 m³ (refurbished)
- Regrind cyclone cluster (refurbished)
- Regrind ball mill 16.5′ × 36.5′ (refurbished)
- Regrind cyclone (refurbished)
- Four first cleaner and four cleaner scavenger flotation cells 8 x 130 m³ (refurbished)
- HIGmill™ 1100 kW (new)
- HIGmill Cyclopac (new)
- Secondary flotation Jameson Cell (new)
- Two third cleaner and scavenger flotation column (refurbished).

17.3.7 Concentrate Thickening and Dewatering

The final flotation concentrate will be collected in a feed pumpbox and transferred to the concentrate thickener and clarifier circuit for solids recovery, with a target of 65% solids. From there, the clarified underflow will be pumped to a new filter press, producing a dewatered concentrate cake. The final product will be discharged to the concentrate stockpile for storage and handling.

Major 3quipment in this area includes the following:

- Concentrate thickener (refurbished)
- Clarifier (refurbished)
- Vertical filter press (new).

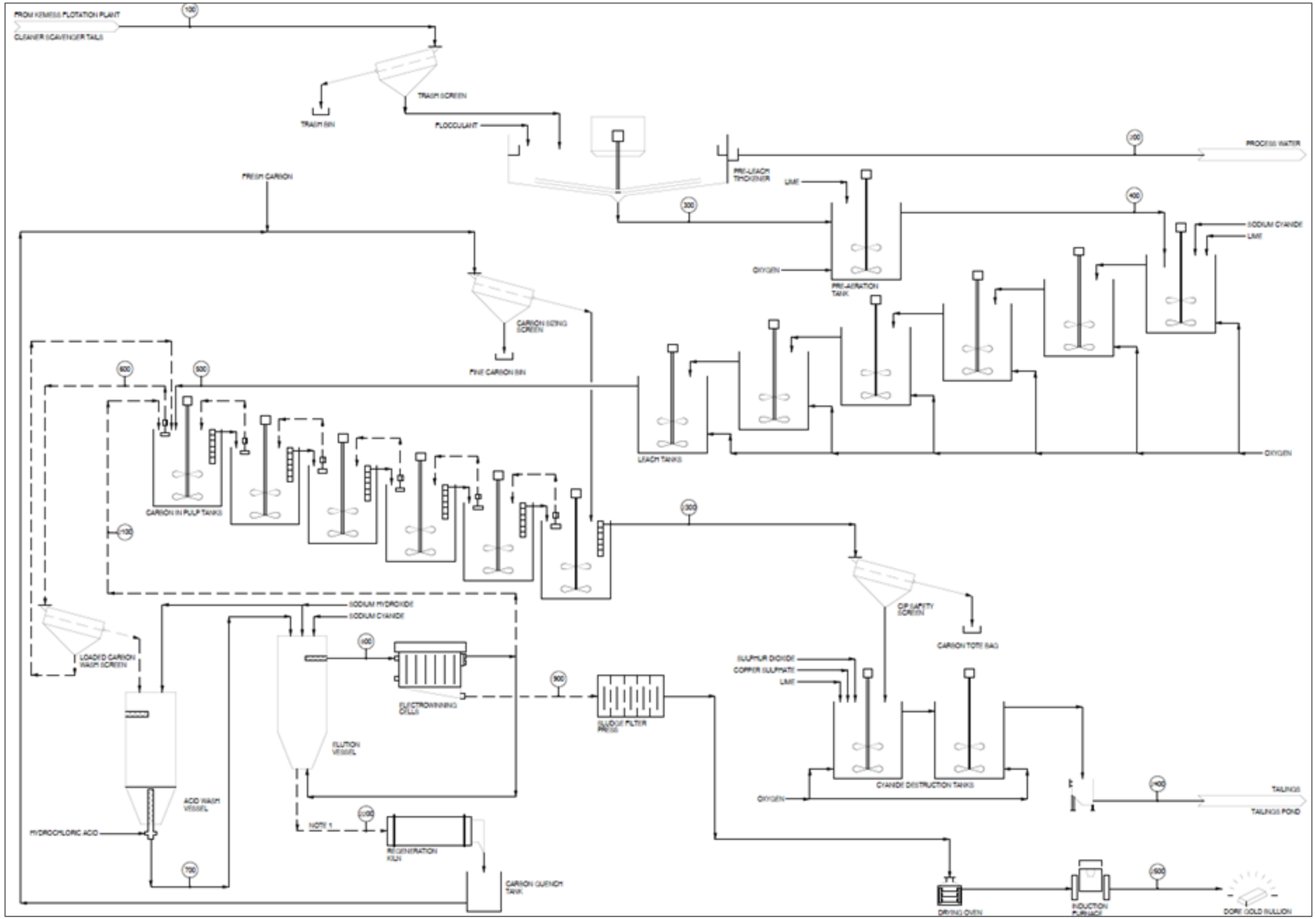
17.4 LEACH PLANT AND ADR

Testing has identified an opportunity to enhance gold recovery by leaching the cleaner-scavenger tails, which contains between 0.49 to 1.15 g/t Au. The feed is pyrite-rich and study work has shown that a conventional cyanide leach with an ADR is the most effective way to improve global gold recovery.

17.4.1 Process Flowsheet

The overall process flow diagram for the leach plant is shown in Figure 17-3.

Figure 17-3: Leach plant process flow diagram



17.4.2 Pre-leach Thickening

Ground slurry from the cleaner scavenger tails reports to a trash screen for removal of oversize material. Screen oversize is rejected to a trash bin, while undersize slurry flows to the pre-leach thickener feed box. The pre-leach thickener has a diameter of 38 m and is designed to increase slurry density and recover process water. Flocculant is dosed to promote solid-liquid separation. Thickener overflow is directed to the process water tank, while underflow at a target density of approximately 50% w/w solids is pumped to the pre-aeration tank.

17.4.3 Leaching

The thickener underflow slurry is pumped to the leaching circuit, which consists of one pre-aeration tank and six leach tanks, each 13 m in diameter and 16 m high, with 1 m freeboard. All tanks are mechanically agitated and arranged in series. The pre-aeration tank is sparged with oxygen to achieve a dissolved oxygen target of 18 mg/L, and lime is added to maintain a pH of 10.5–11. Sodium cyanide solution (30% w/w) is introduced in the first leach tank to initiate gold dissolution. Oxygen sparging continues throughout the circuit to maintain oxidizing conditions. The leach circuit provides sufficient residence time for cyanidation to proceed efficiently before the slurry discharges to the CIP feed launder.

17.4.4 Carbon-in-Pulp

The leached slurry enters the CIP circuit, which consists of six tanks operating in carousel mode. Each tank is 5.35 m in diameter and 5.8 m high. Activated carbon remains in the tanks while slurry is pumped between tanks through interstage screens, enabling countercurrent contact between carbon and pulp. This configuration maximizes gold adsorption onto the carbon phase.

Loaded carbon is periodically withdrawn from the first CIP tank and transferred to the loaded carbon wash screen. Screen undersize returns to the CIP circuit, while oversize loaded carbon advances to the acid wash vessel.

17.4.5 Adsorption, Desorption and Recovery (ADR)

Acid Wash

Loaded carbon batches of approximately 4 t are treated in the acid wash vessel to remove inorganic deposits. A dilute hydrochloric acid solution is circulated through the column, followed by a water rinse and caustic neutralization step. Spent acid and rinse water are discharged to the tailings thickener feed box. After washing, carbon is transferred to the elution column using process water.

Elution

Gold stripping is performed using a high-temperature, pressure process. A barren solution containing NaOH and NaCN is circulated through the elution column at elevated conditions of 135°C and >240 kPa. The solution is heated via heat exchangers and water heaters, and pressure is controlled to prevent boiling. The pregnant eluate exiting the column is directed to electrowinning cells.

Electrowinning and Gold Casting

Pregnant solution from elution is processed in electrowinning cells equipped with cathodes and anodes. Gold-bearing sludge is removed from cathodes using high-pressure water and dewatered in a filter press. The sludge is dried in an oven, mixed with fluxes, and smelted in an induction furnace to produce doré bullion.

Carbon Regeneration and Fines Handling

Stripped carbon is reactivated in a rotary kiln operating at 750°C. Regenerated carbon is quenched in water and screened to remove fines. Oversize carbon returns to the CIP circuit, while fines are collected in a dedicated tank and dewatered in a filter press. Filter cake is bagged for off-site recovery.

Cyanide Destruction

Tailings from the CIP circuit passes through a carbon safety screen. Oversize carbon is collected for recycling, while undersize slurry flows to the cyanide destruction circuit. Cyanide detoxification is performed using a SO_2/O_2 process in two tanks operating in series, providing a total retention time of approximately 3 hours. Liquid SO_2, oxygen, copper sulfate, and lime are added to convert residual cyanide to non-toxic species. Treated slurry will be discharged into the old Kemess South Open Pit for LOM.

17.4.6 Tailings

Tailings piping and tailings embankment construction and management are discussed in Item 18-4.

17.4.7 Reagent Handling and Storage

The reagent handling system will include storage, mixing, and metering equipment as needed for each of the required reagents. Each reagent will be located in a dedicated reagent containment are to prevent environmental contamination and mixing of incompatible reagents. Appropriate ventilation, fire systems, spillage control and safety equipment such as eye wash stations will be located through the area.

Reagent consumptions were estimated based on test work results as described in Item 13-1. A summary of average annual consumption rates for each reagent and operating consumables.

The required reagents are listed in Table 17-2. The system will be refurbished, pending full evaluation.

Table 17-2: Reagents List

Reagent	Use	Handling
Lime (CaO)	pH modifier; depresses pyrite and stabilizes flotation conditions	Received in bulk; slaked and mixed with process water; distributed to grinding, rougher flotation, and cleaner flotation circuits
SEX (Sodium Ethyl Xanthate)	Sulphide collector; promotes flotation of copper and gold-bearing sulphides	Received in 800 kg bags; dissolved in a dedicated mixing tank; added into rougher flotation and cleaner scavenger circuits
Depressant CMC (Carboxymethyl Cellulose)	Depressant; controls gangue entrainment and improves cleaner selectivity	Received solution; prepared as solution with process water; dosed to cleaner and cleaner scavenger flotation circuits
R3477 Activator	Collector/Promoter; enhances recovery of difficult or partially oxidized sulphide minerals	Received in bulk liquid form; stored in a dedicated tank; metered into rougher and cleaner flotation stages
MIBC Frother (Methyl Isobutyl Carbinol)	Frother; controls bubble size and froth stability	Received as bulk liquid; stored in a dedicated frother tank; distributed to rougher flotation, first cleaner, and cleaner scavenger circuits
Flocculant	Settling aid for thickening of concentrate and tailings streams	Delivered as 36 × 25 kg bags on pallets; hydrated and dissolved in process water; transferred to a storage tank and dosed to concentrate and tailings thickeners

Additional reagents will be needed for the leach circuit. All storage and handling will be constructed with the leach facility.

Required reagents for the leach plant are listed in Table 17-3.

Table 17-3: Leach circuit reagents

Reagent	Use	Handling
Hydrochloric acid (32%)	Acid wash reagent used in the activated carbon circuit to dissolve inorganic foulants and restore carbon activity.	Received in bulk liquid form and stored in a dedicated, bunded tank; pumped under controlled conditions to the carbon acid-wash circuit.
Sodium cyanide	Primary gold leaching reagent; forms a soluble gold–cyanide complex in the leach circuit	Received as a concentrated solution or solids dissolved in process water; prepared to plant strength and metered to the leach, cleaner, and cleaner-scavenger circuits under strict safety procedures.
Activated carbon	Adsorbent for dissolved gold cyanide complexes in the carbon-in-pulp (CIP)	Received in solid form in bulk bags; discharged via a bulk-bag breaker to the fresh carbon tank and transferred to adsorption tanks as required.
Sodium hydroxide (50%)	pH modifier in the elution circuit; used to adjust the stripping solution and assist desorption of gold from loaded activated carbon.	Received in bulk liquid form; stored in a dedicated caustic tank; pumped to the elution circuit with appropriate materials of construction and safeguards.
Liquid SO_2	Reducing agent and key reagent in the SO_2/air cyanide destruction process, converting free and weak-acid dissociable cyanide to less toxic species.	Received in bulk liquid vessels; fed via closed piping and flow-controlled metering to the cyanide destruction circuit.
Copper sulphate	Catalyst in the SO_2/air cyanide destruction process, promoting oxidation of cyanide and precipitation of copper as hydroxide.	Received as a bulk solid; dissolved in process water to the required strength, then metered to cyanide destruction tanks.
Anti-scalant	Additive in the elution/ADR circuit to inhibit scale deposition on heaters, pipes, tanks, and vessels, maintaining heat-transfer efficiency and flow capacity.	Supplied as a liquid or solid product; diluted to the recommended dosage strength and dosed continuously to the elution and associated water circuits.

17.5 ENERGY, WATER AND PROCESS MATERIALS

17.5.1 Process materials

Reagent and consumable consumption estimates were developed using anticipated material characteristics, industry benchmarking, and initial testwork results. Table 17-4 presents the projected average annual usage for all reagents and consumables.

Table 17-4: Annual reagent and consumables requirements

Description	Unit	Annual consumption
Grinding Media		
SAG Mill A (5.25” steel balls)	t/a	2,281
SAG Mill B (5.25” steel balls)	t/a	2,281
Ball Mill A (2 – 2.5” steel balls)	t/a	2,099
Ball Mill B (2 – 2.5” steel balls)	t/a	2,099
Regrind Ball Mill (1” steel balls)	t/a	1,197
Regrind HIGMILL - HIG1100kW (2mm T-380)	t/a	123
Reagents		
Lime (CaO)	t/a	13,688
SEX	t/a	365
CMC	t/a	119
R3477	t/a	319
MIBC	t/a	1186
Flocculant	t/a	15
General Mill Operating Supplies	t/a	
Liners		
SAG Mill A (new)	set/a	1.8
SAG Mill B	set/a	1.8
Ball Mill A (new)	set/a	0.6
Ball Mill B	set/a	0.6
Existing Regrind Ball Mill	set/a	0.3
Regrind HIGMILL - HIG1100kW (5000L)	set/a	1

17.5.2 Site Water and Air

Process Water

Process water will consist of concentrate thickener and concentrate clarifier overflow, leach plant feed thickener overflow and reclaim water from various reclaim water sources, including the two tailings ponds. Process water will be stored in process water tanks before being distributed across the plant to various end users.

The system will be refurbished, pending full evaluation.

Fire Water

Fire water for the process plant will be stored in a dedicated volume within the raw water tank. A dedicated pump skid consisting of an electrical pump, jockey pump and diesel pump will supply water from the fire water reserve volume to the distribution system.

The system will be refurbished, pending full evaluation.

Gland Seal Water

Gland seal water for the plant will be sourced from the raw water tank and pumped to the various users across the plant site.

The system will be refurbished, pending full evaluation.

Potable Water

Potable water will be produced by an on-site potable water plant which processes water from the raw water tank.

Processed potable water will be stored in a dedicated storage tank before being distributed to the various end users in the process plant.

The system will be refurbished, pending full evaluation.

Plant Compressed Air

High pressure plant air will be produced by compressors to meet plant requirements. The high-pressure air supply will be collected in a plant air receiver before being distributed across the plant. The system will be refurbished, pending full evaluation.

Power Consumption

The total installed load for the process plant is estimated at 66 MW, with an annual power consumption estimated of 454 GWh/a. Operating costs associated with the plant power consumption are available in Item 21-5.

18 PROJECT INFRASTRUCTURE

18.1 OFF-SITE ROAD AND LOGISTICS

18.1.1 Access Road

The 380 km access road from MacKenzie to Kemess is being maintained and used on a seasonal basis for ongoing site reclamation. Existing forest service Cheni Mine Road is a 16.5km all-season gravel road connecting the Kemess North site to the Kemess Main Road turn off. To accommodate non-loaded mining trucks, two bridges along the Cheni Mine Road require upgrading in addition to minor clearing and widening activities.

All internal roads within the site required to connect the facilities, and ensure the separation of light and heavy vehicles, have been maintained and will be used during the Kemess restart operation.

18.1.2 Air Strip

A 1.6 km all-weather air strip capable of handling most Short Takeoff and Landing (STOL) aircraft and transport planes when heavier loads are required. A twin-engine Beech aircraft was routinely used to ferry personnel on and off site during the operation of the Kemess South Mine. The airstrip could accept Dash 8 aircraft and heavy haul type STOL aircraft during past operations. The strip continues to be maintained and used by the care and maintenance personnel as their main site access route.

18.1.3 Mackenzie Rail Loadout Facility

Next to a rail spur, Centerra retains a trans-shipment facility at MacKenzie for shipment of Mount Milligan concentrates which will also accommodate Kemess shipments. Grinding media consumables are received by rail in MacKenzie from various suppliers.

During initial construction, oversized loads were transported by rail to avoid highway size and weight restrictions and rail transport will again be utilized for the Kemess Restart Project when necessary. The loadout facility at Mackenzie will need to be upgraded to be able to simultaneously service the Kemess mine and the Mount Milligan mine.

18.2 SITE INFRASTRUCTURE

18.2.1 Existing Kemess South Site Infrastructure

The Kemess South Mine site previously supported a 50,000 t/d open pit mining and milling operation. The infrastructure from the previous 50,000 t/d open pit mining and milling operation is largely intact and

is currently being maintained by onsite care and maintenance personnel and remains adequate for the Kemess restart project. This includes:

- Administration Facilities: The administration complex consists of established and maintained office buildings, mine dry facilities, warehousing and a surface workshop. Administration office spaces from the previous operation are available and sufficient for the proposed Kemess restart operation.
- Assay and Metallurgical Laboratory: The assay laboratory is located within the process plant and will be refurbished pending full evaluation. The lab is comprised of a storage area, office stations, scale room, AA room, wet lab and met labs and fire assay. This building is equipped with fire protection and an alarm system. The laboratory requires bottled nitrogen and hoods with ventilation.
- Truck Shop: A truck shop equipped with truck bays, ancillary equipment bays, wash bays, tire change area, welding area, lubrication storage area.
- Security facilities: Infrastructure to ensure the safety of personnel and assets on and off site including the site gate house, and security fencing.

No new infrastructure buildings are planned for the project in the Kemess South Site.

18.2.2 Kemess North Site Infrastructure

New infrastructure, including the construction of an infrastructure pad, will be required at the Kemess North site to support mine operations. This infrastructure will include:

- Mine Administration Facilities: All the facilities that form the mine infrastructure area; truck bays (day to day service only; overhauls to be done at the South truck shop), ancillary equipment bays, wash bays, tire change area, welding area, lubrication storage area, diesel storage and distribution, offices, and warehouse.
- Communications: The network communication architecture, energy management system, and associated hardware to support the operation.
- Mobile Equipment Shop: The mobile equipment required to support the operation.
- Security Facilities: Infrastructure to ensure the safety of personnel and assets on and off site including the site gate house, security fencing.
- Information Technology: The information technology (IT) and communications requirements for the project including data, media, and voice transmission services and closed-circuit television (CCTV) systems.
- Emergency facility with mine rescue station and ambulance vehicle.

18.2.3 Paste Backfill Plant

The backfill system proposed for Kemess Mine includes additional thickener tanks and piping infrastructure from the mill to deliver thickened tailings to the paste fill plant which will be located, along with a binder storage tank, at the South Portal access to KUG. An underground booster system is planned to be constructed in the decline to pump the paste fill through the 3.6 km decline to the underground mine distribution system.

Backfill Plant Process Description and Major Equipment

For the purpose of the PEA, the paste plant is assumed to be located on surface near the underground access portal, and the tailings will be filtered at the paste plant. The following system description is based on these assumptions.

The tailings stream will be a blend of predominantly open pit tailings with a smaller proportion of underground tailings. The blended tailings are assumed to be thickened at Pump House No.1 prior to long-distance pumping in order to reduce volumetric flow rates and associated pipeline sizes. Binder will be delivered directly to the paste plant by bulk transport trucks. The paste plant will filter the thickened tailings and blend the filtered tailings with water and binder to produce the paste. The paste is assumed to be pumped underground using a positive displacement pump.

18.2.4 Camp and Accommodations

The existing camp at Kemess South includes a kitchen, approximately 150-person bunkhouse units, a potable water treatment facility, a sewage treatment facility, backup generator sets, site services maintenance shops, and recreational facilities. The camp was previously used to support Kemess South operations and has been maintained in operable condition.

The camp has been inspected and assessed as suitable to support the study and construction phases of the Project; however, it is not expected to be adequate for the full life of mine. Capital costs have been included for the renovation and expansion of camp services as well as the development of a new camp to support operations for the life of mine. Renovation costs have been estimated and included as capital costs in the study.

18.2.5 Power and Electrical

Power Line

Electrical power to the site is supplied by a 380 km, 230 kV transmission line extending from the BC Hydro Kennedy Substation to the mine site, including associated step-down transformers, backup diesel generators, and supporting infrastructure required to power the process plant and the Kemess South site. Centerra owns the transmission line and all related on-site electrical infrastructure.

The transmission line is currently energized and in use, providing power for Kemess South during the care and maintenance period. Annual inspections are up to date, and the line is in good operating condition. The main substation supplying the Kemess transmission line will require upgrades to support power delivery to both the Kemess Project and the Mount Milligan mine. Engagement with BC Hydro is ongoing to confirm upgrade requirements and to secure sufficient electrical power capacity for the Project.

Site Power Distribution

Medium-voltage power at 13.6 kV is distributed from the Kemess South Process Plant to substations across the site, including the Kemess Main Zone Open Pit and the pumphouses servicing the TSF. Site power distribution infrastructure includes facility-level transformers, low-voltage distribution systems, electrical control rooms, and associated ancillary equipment required to support operations.

A new 13.8 kV distribution line will be constructed to supply power to the Kemess North and underground mining areas. The final routing of this line will be determined in the next phase of studies and is expected to generally follow the conveyor alignment.

At the mine site and along the conveyor corridor, 13.8 kV/600 V transformers will be installed to step down the distribution voltage and supply power to process facilities, infrastructure, and other operational areas of the site.

18.3 PROCESS PLANT

The existing ore processing facilities previously processed in excess of 50,000 t/d of ore with mineral characteristics similar to those of the Main Zone deposit. The plant is currently maintained by care and maintenance personnel. Basic fire suppression systems remain operational, and the mills are secured on cradles. One grinding line has been removed, however, the remaining grinding line is planned to be used for the Project, and the balance of the process plant facilities and equipment remains intact.

Additional details regarding the process plant are provided in Item 17 of this Technical Report.

18.4 TAILINGS STORAGE

18.4.1 Background

The Kemess Mine operated from May 1998 to March 2011, during which time approximately 213.4 Mt of tailings were stored at KS TSF, with an additional 17.4 Mt of tailings directed to the mined-out portion of the KS Open Pit (KUG TSF) towards the end of operations. The KS TSF was constructed as a modified centreline embankment over 12 stages between 1996 and 2010 as shown in Figure 18-1 and Figure 18-2. The original design was completed by KP, who oversaw the construction of Stage 1 and

Stage 2. Subsequent stages were designed and monitored by Wood. After operations ceased, reclamation and closure activities, including the construction of a spillway, were carried out.

Figure 18-1: Typical section through KS TSF main dam

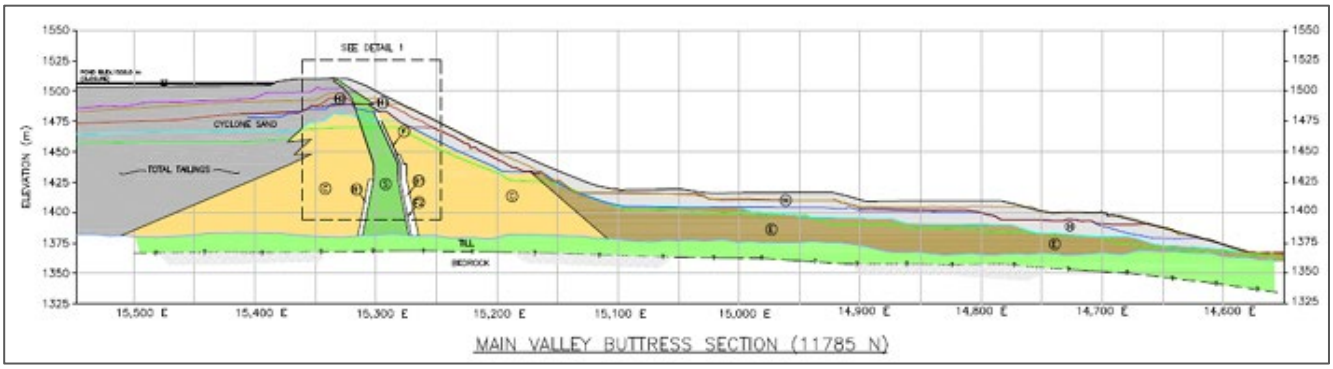
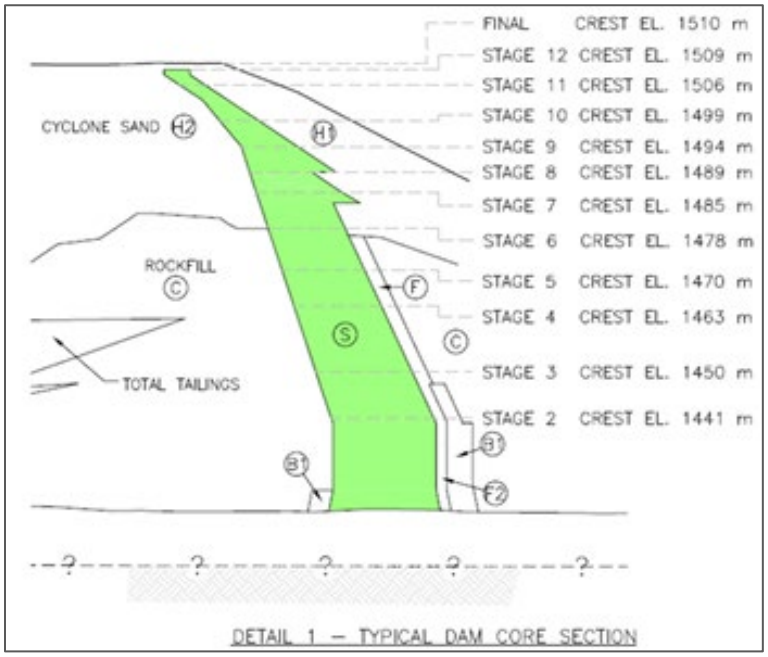


Figure 18-2: Detailed section of KS TSF main dam



A 2025 study contracted to a specialist engineering consultant focused on identifying and designing suitable tailings storage solutions utilizing the existing Kemess South (KS) Tailings Storage Facility (TSF) and the Kemess Underground (KUG) TSF for the deposition of an additional 261 Mt of tailings and 158.5 Mt of potentially acid-generating (PAG) waste rock.

A preliminary tailings management plan has been developed to support the PEA for the restart of operations at the Kemess Mine. The study focused on evaluating the potential of reopening the KS and KUG TSFs to manage tailings based on the updated mine plan. The study included the following:

- Development of a conceptual deposition plan
- Preliminary design and stability assessments for the KS TSF embankments

- Review of KS TSF diversion system and design of alternative diversion system
- Preliminary cost estimates for the KS and KUG TSFs; and
- Develop closure concept for the KS and KUGTSFs

18.4.2 TSF Strategy

An assessment was undertaken by an independent consultant to determine the storage capacity of the Kemess South Tailings Storage Facility (KS TSF), based on the following criteria:

- Storage of approximately 15 Mt of tailings per year. The predicted annual volume is comprised of cleaner tailings, cyclone overflow, and bulk rougher tailings; and excludes cyclone sand underflow required to raise the Main Dam.
- Deposition and storage of approximately 5.5 Mt of Potentially Acid Generating waste rock per year.
- A maximum of 22% of the scavenger tailings volume (by mass) is available for dam construction material, via cycloning. Cyclone operations will be limited to 9 months per year at an efficiency of 85%, resulting in the delivery of approximately 2 Mt of cyclone sand underflow per year for embankment and buttress construction.
- Filling of the TSF and construction of the Main Dam will maintain a material balance.
- A freeboard allowance of 5 m between the tailings surface and the Main Dam crest elevation to account for sloping tailings beaches.
- The dam elevation is constrained by the topographical low spot of the ridge between the Main Dam and the South Diversion Dam (SDD) (approx. EL. 1,580 masl) (Figure 18-3).

The existing KS TSF embankment will be raised 26 m to a final elevation of 1,537 masl to store approximately 122 Mm^3 of operating pond, and 3 m freeboard. The KS TSF main embankment will be raised using the centreline construction method, with a low-permeability glacial till keyed into the existing dam. The KS TSF main embankment downstream shell and stabilizing buttress will be comprised of cyclone sand produced from rougher tailings.

The KUG TSF will utilize the existing Kemess South open pit to store approximately 63 Mm^3 of tailings, a 4 Mm^3 operating pond, and a 3 m freeboard below the existing pit rim elevation of 1,260 masl. The KUG TSF design includes an East Dam and a South Saddle Dam, each constructed to an elevation of 1,295 masl.

18.4.3 Previous Studies

A feasibility-level design for the KUG TSF was completed in 2012 by AMEC (AMEC, 2012). The feasibility level design was based storage of 102 Mt of tailings and 2 Mt of waste rock over a 13-year mine life, with all storage directed to the KUG TSF. The design incorporated a dam on the east side of the pit with a crest elevation of 1,275 masl to provide sufficient storage capacity.

In 2015, a Tailings Alternatives Study was undertaken by AMEC Foster Wheeler (AMEC FW) as part of the Environmental Assessment process. This study evaluated alternative technologies, siting options, and developed a water balance for the KUG TSF. The study considered a total of 27 potential options for the TSF design determined that the KS open pit, coupled with an east dam using conventional tailings, to be the most viable solution based on technical aspects. From 2015 to present, additional studies have been conducted to refine the KUG TSF design. These studies include assessments related to water treatment and management, material parameter characterization, laboratory testing for tailings, and evaluations on increasing the facility's total storage volume.

An Opportunity Framing Study was completed by Klohn Crippen Berger (KCB) in 2020 to explore options for increasing the storage capacity of the KUG TSF. The study evaluated several development alternatives, evaluating different facility geometries and tailings methodologies. A feasibility study was completed in 2020 for the proposed South Saddle Dam, which would be required along the southern perimeter of the pit to support this increased capacity.

In May 2025, KP conducted an assessment to evaluate preliminary tailings and waste management assessment utilizing the KS TSF and KUG TSF, to accommodate all PAG waste rock and tailings storage. The KUG TSF design included an East Dam and a South Saddle Dam, each constructed to an elevation of 1,295 masl, and KS TSF raise to elevation 1,537 masl with a buttress raise and extension.

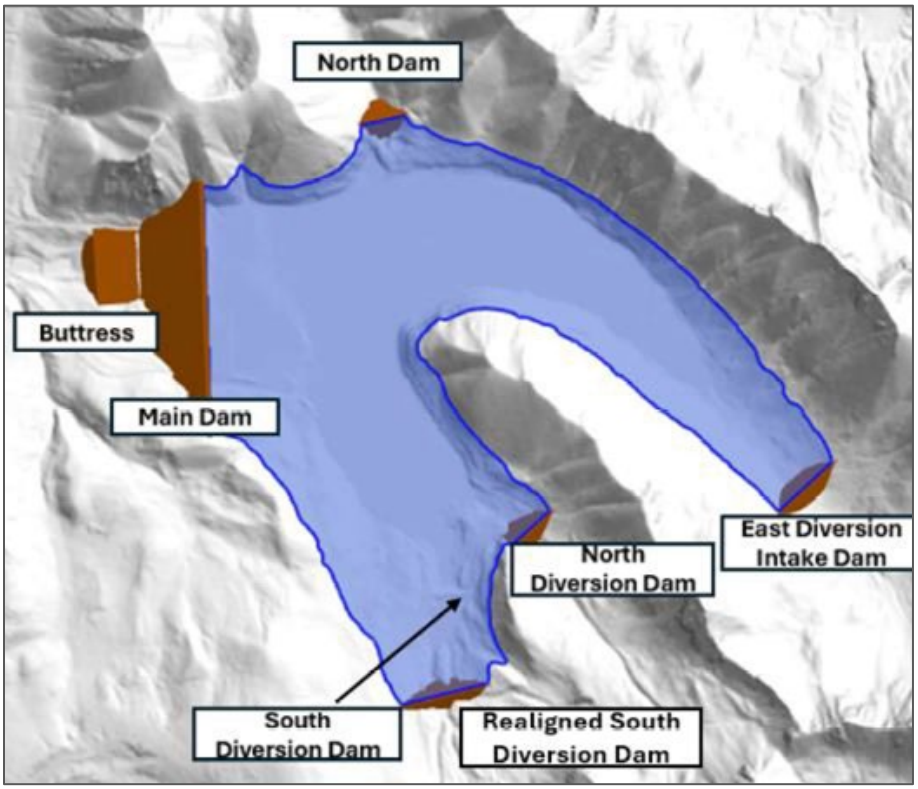
18.4.4 2025 PEA TSF Design

A new North Saddle Dam will be required along the north side of the facility to provide adequate containment. This structure will be constructed as a downstream raise rockfill shell dam with a low-permeability core. The increase in the facility size has required an upgrade to the existing KS TSF diversion system. This upgrade includes raising the existing South and North Diversion Dams using downstream and centreline methods, respectively, as well as adding a centreline raise to the East Diversion Dam. All dams will consist of rockfill shells with low-permeability cores and appropriate transition materials on both sides.

In addition, new South and North Diversion Channels and an East Diversion Conduit will be constructed, all graded to convey flows from the East and North Diversions toward the South Diversion, where they will be discharged to Kemess Creek.

A preliminary layout of the Kemess South TSF facility is shown in Figure 18-3.

Figure 18-3: Modelled KS TSF



18.4.5 Topography

The topography of the site is highly variable and can be divided into three primary zones (AMEC, 2012):

- The KS TSF is located in a high, U-shaped hanging valley within the South Kemess Creek area.
- The narrow valleys of Kemess Lake and North and South Kemess Creeks, where numerous glacial terraces and aggregate deposits are found.
- The broader valley with outwash deposits associated with Attichika Creek.

The elevations across the historical mine site range from 1,150 m to 1,600 m, interacting with a diverse landscape influenced by glacial processes. (AMEC, 2012)

The site topography reflects significant glacial erosion and deposition, characterized by periglacial, glaciofluvial, and glaciolacustrine landforms. More recent deposits consist of fluvial, colluvial, and organic materials. (AMEC, 2012)

The terrain features moderate relief, with elevations increasing toward the north. Most mountains below an elevation of 1,800 masl have rounded summits, whereas higher peaks display distinct features of alpine glaciation, including cirques, arêtes, and horns (AMEC, 2012).

18.4.6 Geotechnical Conditions

Regional Geology

The Kemess property is located at the northeast margin of the Stikine arc terrane, which spans much of western BC, and at the southern end of the Toodoggone mining district, a 100 km long and 30 km wide NNW-trending belt of mineral deposits and prospects in Mesozoic era volcanic rocks and coeval intrusive rocks. The area is known for being prospective for porphyry copper-gold deposits and low sulphidation epithermal gold-silver vein deposits. Within the Kemess property, there are several known porphyry copper gold deposits including Kemess South, Main Zone (which includes Nugget) and Kemess East, as well as several other mineral prospects and showings (AMEC, 2012).

Surficial Geology

The surficial geology in the vicinity of the KS TSF is generally as follows:

- Bedrock is overlain by very dense, well graded basal-type glacial till, which becomes coarser with depth. This deposit thins out at higher elevations along the upper slopes adjacent to the valley.
- Varved glaciolacustrine sediments, comprising of interbedded silt, clay, and thin sand laminations overlay the dense basal glacial till. These deposits are extensive within the South Kemess Creek valley, occurring up to an elevation of 1,410 masl.
- Above the glaciolacustrine sediments is a second basal till layer comprised of well-graded, silty sands and gravels. This layer is thicker on valley slopes while thinning out in the valley bottom.
- Glaciofluvial deposits are found at surface including poorly graded gravelly sands on the north side of the valley.

The surficial geology in the vicinity of the KUG TSF is generally as follows:

- Bedrock is overlain by a basal till that ranges from less than 1 m on steep northern slopes of the pit to more than 10 m in the mill site area.
- Sandy, gravelly glacial veneers, varying from 1 m to 4 m thick, overly the basal till surface and form trending ridges.
- Gravel deposits are present at the outlets of South Kemess Creek and along Kemess Creek, forming terraces above the incised present-day creek course.
- The Attichika Valley south of the camp consists of a wide fluvial plain with terraces.

- Colluvial and fluvial deposits are present on the mid to upper slopes of the South Kemess Creek Valley, while numerous shallow organic deposits have developed in local depressions throughout the region.

Property Geology

The bedrock underlying the Kemess Project area predominantly consists of basal members of the Takla Group, which include green pyroxene-porphyritic lapilli tuffs, flow breccias, and subordinate plagioclase bearing mafic crystals with lithic tuffs. These rocks serve as the host for the intrusive-related Kemess South, Main and Nugget gold-copper porphyry deposit.

Sedimentary units of the Takla Group, primarily consisting of chert, mudstone, and minor limestone, have also been identified on site. These sedimentary units occur as east-west trending bands and are primarily located north of the Kemess South porphyry deposit.

18.4.7 Seismicity

Seismic activity in the Kemess Mine region is historically low. The most significant recorded event occurred in 1969, approximately 30 km northeast of the site, with a magnitude of Mw 5.0. (KCB, 2004). The 2020 National Resources Canada (NRCAN) National Building Code of Canada (NBCC) peak ground acceleration (PGA) values for the site are summarized in Table 18-1.

Table 18-1: Seismic criteria and 2020 NBCC PGA values

AEP	PGA values
100	0.007
475	0.022
1,000	0.037
2,475	0.070
10,000	0.176

18.4.8 Design Basis - General

The Kemess Mine restart scenario under consideration contemplates a one-year ramp-up period. Tailings will be produced each year throughout the mine life. The estimated total tailings production quantities and schedule are summarized in Table 18-2.

Table 18-2: LOM annual tailings production

Mine year	Annual Tailings		
	Rougher (Mt)	Cleaner (Mt)	Total (Mt)
1	11.8	2.1	13.9
2	15.2	2.7	17.9
3	15.5	2.7	18.2
4	15.5	2.7	18.2
5	15.5	2.7	18.2
6	15.5	2.7	18.2
7	15.5	2.7	18.2
8	15.5	2.7	18.2
9	14.8	2.6	17.4
10	15.7	2.7	18.4
11	15.5	2.7	18.2
12	15.5	2.7	18.2
13	15.5	2.7	18.2
14	15.5	2.7	18.2
15	9.3	1.6	10.9
Totals	**221.9**	**39.2**	**261.0**

Notes:
1. Quantities based on 080725 Kemess PEA Mine Plan Summary.xlsx mine plan (AuRico, 2025a).

The general tailings management design assumptions for the assessment are as follows:

- All tailings will be stored within the KS and KUG TSFs.
- No additional storage facilities will be developed.
- Rougher tailings are geotechnical and geochemically suitable for cyclone sand production and available for construction materials.

18.4.9 Hazard Consequence Classification

Hazard consequence classification for both KUG and KS Facilities were evaluated in accordance with the 2019 CDA Mining Dams Bulletin, CDA guidelines and the Code (CDA, 2019; CDA, 2013; EMLI, 2024). The consequence classification of the KUG TSF is considered to be Significant based on potential overtopping of the open pit releasing diluted contaminated mine water to the receiving environment. There is no potential loss of life, the expected economic loss is moderate, and the potential inundation area would be limited with potential for restoration. The consequence classification of the KS TSF Main Embankment is rated as Very High due to the impacts to the sensitive water course of the Kemess Creek valley and the personnel working on and around the downstream area of the KS TSF Main Dam which could exceed 10 but likely be less than 100.

18.4.10 Tailings Characterization

The milling operation will produce two tailings streams from conventional milling of copper and gold ore. Mill throughput is anticipated to be 50,000 tonnes per day (tpd). Tailing streams will be piped and deposited into the designated TSF. Two types of tailings have been designated for the project, rougher tailings and cleaner tailings. Rougher tailings are expected to comprise approximately 85% of the total tailings volume and are anticipated to be non-acid generating (NAG). The remaining 15% are classified as cleaner tailings and are expected to be potentially acid generating (PAG).

A preliminary cyclone sand simulation was conducted to support this study, using the rougher tailings gradation data provided by AuRico in 2025 (AuRico, 2025b). These gradations were provided to a cyclone sand equipment manufacturer Weir to produce a final product with an underflow (sand fraction) material containing less than 15% fines. It is common industry practice to use cycloned sand with less than 15% fines in embankments, based on the performance history of similar structures. This fines content is typically low enough to ensure sufficient permeability, allowing effective drainage and reducing pore water pressures within the dam. Additional laboratory testing of cycloned sand may help optimize the fines content. The results from Weir indicate the rougher tailings are capable of being separated into 30% by mass underflow and 70% overflow (fine fraction) (Weir, 2025). Further test work and updated gradations of the tailings are recommended to refine these results and verify assumed design parameters.

The cyclone sand operation is anticipated to occur for approximately six months per year to meet material balance for embankment construction at an efficiency rate of 85%. The tailings production materials were assigned the following dry densities:

- Cleaner Tailings: 1.3 t/m^3
- Rougher Tailings: 1.3 t/m^3
- Rougher Tailings Underflow (UF)
- Cyclone Sand: 1.7 t/m^3
- Rougher Tailings Overflow (OF): 1.25 t/m^3.

18.4.11 Freeboard

KP conducted Inflow Design Flood (IDF) and wave run-up assessments for both the KS and KUG TSFs under operational conditions. The IDF assessments considered return period precipitation and PMP depths derived from the MetPortal tool for a 72-hour duration event and incorporated climate change considerations applicable to the operational phase of each facility. The minimum freeboard allowances determined for the IDF and wave run-up events are summarized Table 18-3.

Table 18-3: IDF and wave run-up minimum dam freeboard

Facility	Available freeboard (m)	Operation IDF minimum freeboard (m)	Closure IDF minimum freeboard (m)	Wave run-up freeboard (m)
KUG TSF		1.3	0.9	1.6
KS TSF		1.5	1.4	2.1 Embankment slopes 0.5 Tailings slopes

Notes: (1) The operational IDF event is based on the 72-hour design storm event. (2) The closure IDF event is based on the 24-hour design storm event. (3) Wave run-up for the KS TSF Main Dam was assessed assuming shallow-sloping tailings beach slopes and a north–south fetch length (4) Wave run-up for the KS TSF Diversion Embankment was assessed assuming embankment slopes and a west–east fetch length.

18.4.12 Tailings Management Plan

General

The development of the tailings management plan for the Kemess Restart involved an iterative process aimed at optimizing the use of the KUG TSF and KS TSF while maintaining operational integrity. Throughout this process, various deposition scenarios were evaluated, taking into account factors such as facility size, embankment height and geometry, operational efficiency, material balance, and tailings types/technologies. The tailings management plan was ultimately refined and finalized to best meet storage and operational objectives.

Waste Management Plan

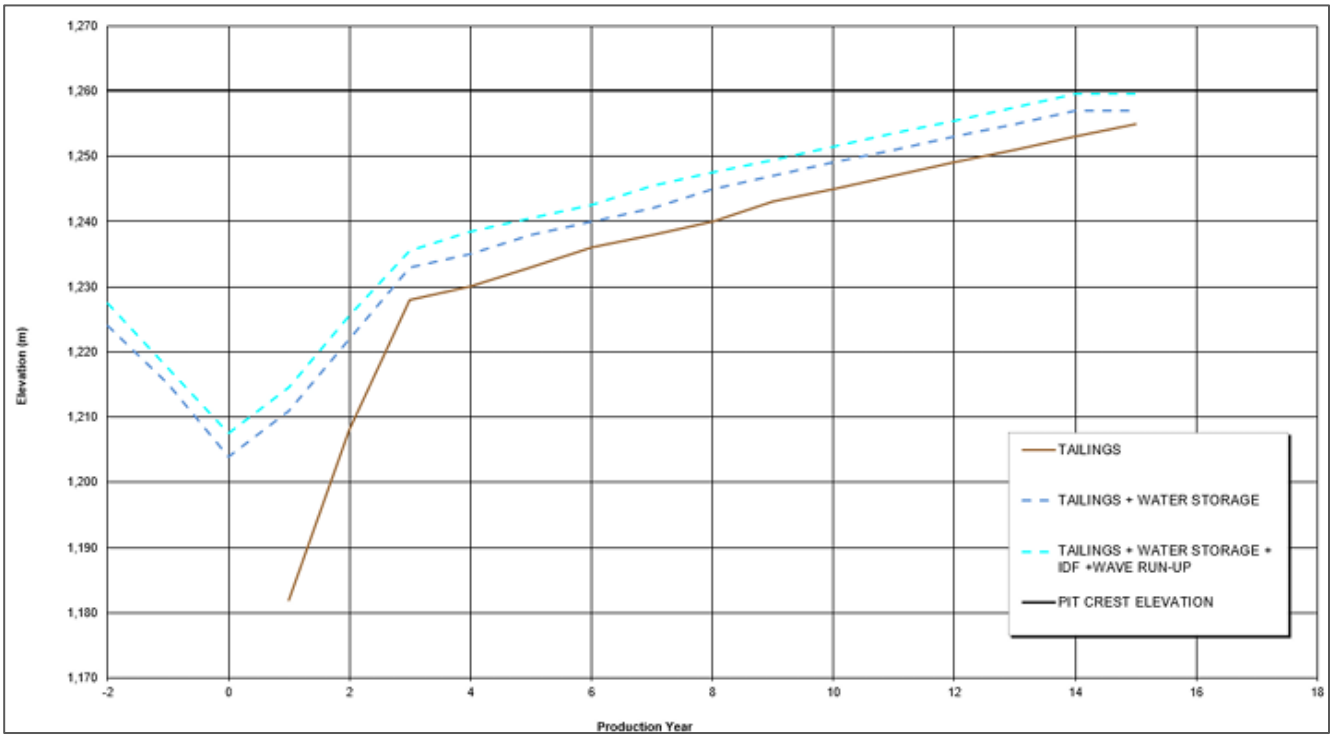
The general tailings deposition plan involves depositing both cleaner and rougher tailings material into the KUG TSF until Year 3. In Year 4, the rougher tailings will be redirected to the KS TSF, while cleaner tailings will continue to be deposited in the KUG TSF until the end of operations. In Years 4 to 15, rougher tailings deposition will occur in the KS TSF, utilizing a double cyclone method to produce underflow tailings containing less than 15% fines, which will be utilized as embankment construction material. For the purposes of this assessment, the cyclones are assumed to operate at 85% efficiency for six months of the year, resulting in a material balance of cyclone sand suitable for embankment and buttress construction. The detailed tailings management strategy for each facility is outlined in Table 18-4 and Table 18-5 with filling curves provided in Figure 18-4 and Figure 18-5.

Table 18-4: KUG TSF tailings management strategy

Operating Year	Tailings volumes		
	Rougher (Mm^3)	Cleaner (Mm^3)	Total (Mm^3)
1	9.1	1.6	10.7
2	11.7	2.1	13.8
3	11.9	2.1	14.0
4		2.1	2.1
5		2.1	2.1
6		2.1	2.1
7		2.1	2.1
8		2.1	2.1
9		2.0	2.0
10		2.1	2.1
11		2.1	2.1
12		2.1	2.1
13		2.1	2.1
14		2.1	2.1
15		1.3	1.3
Totals	**32.7**	**30.1**	**62.8**

Notes: (1) Cleaner and rougher tailings until Year 3, then only cleaner tailings. (2) Quantities based on 080725 Kemess PEA Mine Plan Summary.xlsx mine plan provided by AuRico July 2025.

Figure 18-4: KUG TSF filling curve



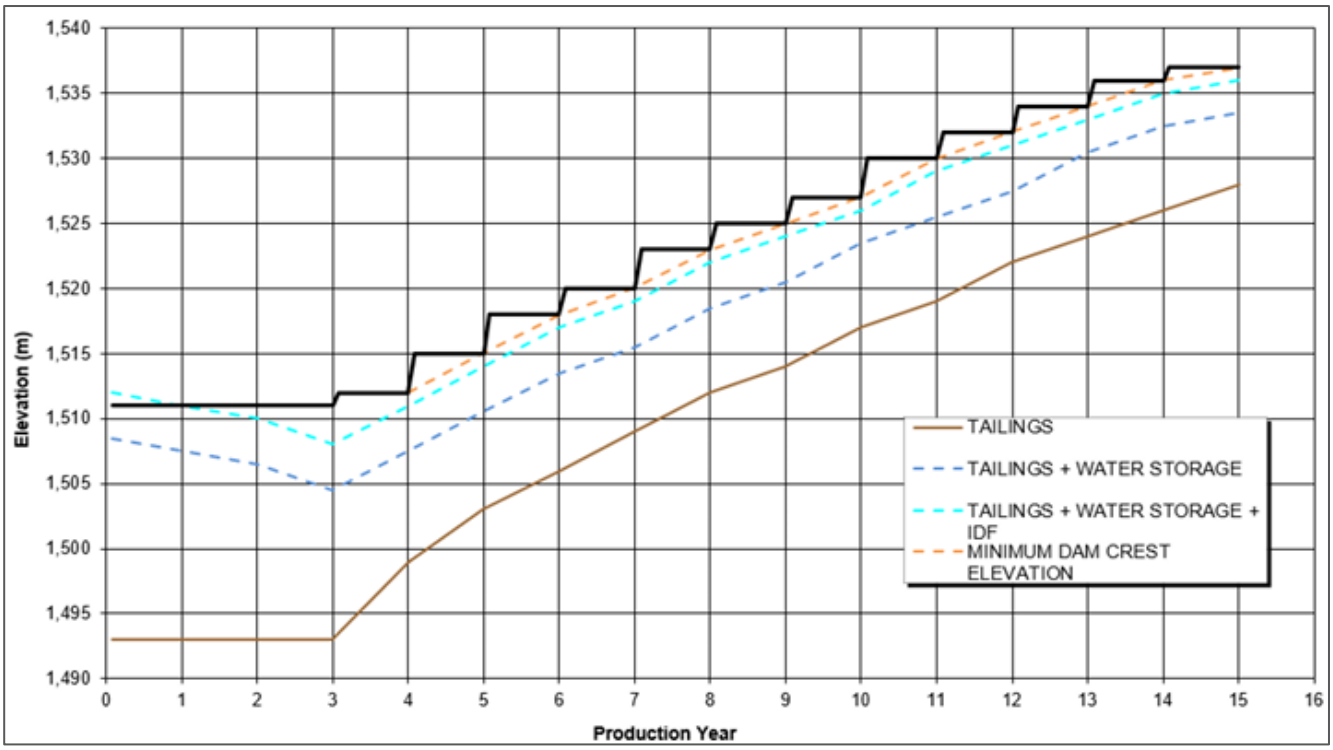
Notes: (1) In Years -2 and -1, the KUG TSF Pond will be drawn down 7 Mm^3 to an EL 1205 masl and operating pond size of 16 Mm^3 prior to operations. (2) In Years 1 to 3, an additional 6 Mm^3 will be required to be drawn down during operations to achieve an operating pond of 4 Mm^3 by end of Year 3.

Table 18-5: KS TSF tailings management strategy

Operating year	Rougher tailings bulk (Mm^3)	Rougher tailings overflow (Mm^3)	Total (Mm^3)	Rougher tailings underflow to embankment (Mm^3)
4	6.9	3.7	10.6	1.2
5	6.9	3.7	10.6	1.2
6	6.9	3.7	10.6	1.2
7	6.9	3.7	10.6	1.2
8	6.9	3.7	10.6	1.2
9	6.5	3.5	10.1	1.1
10	6.9	3.7	10.7	1.2
11	6.9	3.7	10.6	1.2
12	6.9	3.7	10.6	1.2
13	6.9	3.7	10.5	1.2
14	6.9	3.7	10.5	1.2
15	4.1	2.2	6.3	0.7
Total	**79.3**	**42.7**	**122.0**	**13.4**

Note: (1) Quantities based on 080725 Kemess PEA Mine Plan Summary.xlsx mine plan provided by AuRico July 2025.

Figure 18-5: KS TSF filling curve



Note: (1) KS TSF pond will be drawdown to from 37 Mm^3 to pond volume of 25 Mm^3 (EL 1504 masl) prior to operations.

Deposition Plan

Tailings slopes were determined based on the 2011 bathymetric surveys (AMEC, 2012) and operational experience input from AuRico. For the purposes of this assessment, the ranges were refined to specific values. These slopes are preliminary and will be further refined in future studies.

- Sub-aerial rougher tailings discharge:
  - Beach slope above water surface: 1%
  - Beach slope below water surface: 0.5%.
- Sub-aqueous cleaner tailings discharge: 0%.

A proposed project development sequencing is described as follows:

- Pre-production: Establish access and installation of project infrastructure; tailings pipelines and reclaim system. Drawdown KUG pond to El. 1,205 masl.
- Year 1 through 3: Tailings deposition will commence in the KUG TSF. The deposition will be conducted sub-aqueously. Deposition location will be relocated periodically for water and tailings management purposes.
- Years 4-15: Subaqueous deposition of cleaner tailings in the KUG. Rougher tailings will be redirected to the KS TSF. Deposition at the KS TSF will occur from the Main Dam crest, with cycloning used to produce underflow material for placement on the downstream side of the dam, while overflow and bulk rougher tailings will be discharged to the TSF basin. Downstream placement of cyclone sand will require active management to ensure the embankment is constructed to the design slope of 3H:1V. In addition, management of tailings beach development will be required to maintain a beach length of 500 m to supporting efficient tailings deposition throughout operations.

Pond Volume

A water balance model is currently being conducted by Environmental Resources Management Consultants Canada Ltd (ERM) as a separate assignment.

For this tailings management strategy, it is assumed that in pre-production years, the KUG TSF pond will be drawn down to elevation 1,205 masl to reduce the volume of water requiring treatment from the facility. It is assumed that this drawdown pond and then maintaining a pond volume of approximately 4 Mm^3 will be sufficient to provide coverage over cleaner tailings throughout deposition and ensure adequate water availability for operation of the KUG TSF.

The KSF will require a pond volume of approximately 25 Mm^3 to support operations based on the proposed location of the reclaim system and the deposition strategy. This will require the current pond

to be drawn down to approximately 1,505 masl prior to the start of operations. The closure pond will be maintained at a volume of 25 Mm³ to provide a water cover over the majority of the tailings beaches and to satisfy spillway design requirements.

18.4.13 KUG TSF Design

Overview

The KUG TSF is developed from the existing open pit operations and will require minimal modifications to its current configuration. Tailings and water management will be conducted through a barge system that transfers water from the pit to the plant site. Tailings lines will be routed from the mill to the TSF and deposited within the pit via subaqueous deposition. Tailings and reclaim systems will target different areas to keep streams separated. The rate of rise in the pit is expected to be approximately 10 m in elevation per year for the first three years, followed by an average of 2 m per year until the end of deposition.

Roads and Additional Infrastructure

Existing roads will remain functional, with additional access roads reinstated as required to allow for tailings pipeline installation for tailings deposition within the facility.

Closure Design

The objectives of the closure plan for the KUG TSF are to restore the site to as natural a state as practicable, protect the downstream environment, and effectively manage surface water. The preliminary closure concept includes the construction of an emergency spillway at the eastern low point of the facility. The spillway will maintain a consistent water level within the TSF while allowing surplus water to be safely released to the environment. The conceptual spillway has been designed to safely convey the Probable Maximum Flood (PMF) with an inflow rate of 27 m^3/s. The design features a 3 m deep trapezoidal channel with a 5 m base, 2H:1V side slopes, and an average longitudinal grade of 4%, directing flow northeast toward Kemess Creek.

Water quality modelling is recommended to determine whether active water management or treatment will be required during closure. Additionally, water balance modelling is required to further refine and confirm the spillway design parameters, invert and cover requirements.

18.4.14 KS TSF Design

Existing Design

The KS TSF currently consists of the Main Dam, South and North Diversion Dams (SDD and NDD), and East Diversion Intake. The Main Dam is a 135 m high zoned earth-fill, modified centreline construction dam composed of a low permeability core, filter zones, and a rockfill and cyclone sand dam shell.

Construction was carried out in stages utilizing local borrow materials, rockfill from the open pit, and cyclone tailings sand. The Main Dam was completed to its current crest elevation of 1,511 masl in 2007. The SDD and NDD are zoned earth-fill embankments with a low permeability core completed to crest elevations of 1,515 masl and 1,540 masl respectively during the initial phases of construction. The East Diversion Intake, NDD and SDD divert water around the TSF to South Kemess Creek via a network of diversion conduits.

A Sedimentation Pond and a Seepage Recycle Pond located at the downstream toe of the Main Dam collected and managed bleed water from cyclone underflow placement, dam seepage, and runoff for recycle to the TSF during operations. These facilities currently passively discharge to South Kemess Creek.

Embankment Design

The expansion of the KS TSF Main Dam will consist of raising the main embankment to an elevation of 1,537 masl. The existing buttress at the base of the dam will need to be extended by 50 m and raised to elevation 1,417 masl as shown on Drawing C0210 to meet stability requirements. The dam will be constructed as a centreline raise with a downstream slope of 3H:1V. The vertical raise will include a central glacial till core keyed into the existing crest of the dam. Cycloned sand underflow material containing less than 15% fines will be used to supplement construction material for the Main Dam and buttress.

The facility will also consist of three diversion dams that divert catchment flows along the east side of the TSF, as well as one saddle dam on the north side to provide containment. These structures are further described in the following sections.

The Sedimentation Pond and Seepage Recycle Pond from previous operations will be reinstated to manage bleed water from cyclone underflow placement, dam seepage, and runoff from the TSF, with recovered water recycled back to the TSF.

North Saddle Dam

A saddle dam will be required along the north side of the TSF in Year 7 (EL 1,520 masl) to contain tailings and water within the TSF. This new structure, referred to as the North Saddle Dam, will be constructed as a zoned earth-fill embankment with a low-permeability core and built using a downstream raise construction method, as shown on Drawing C0211.

Diversion System

Overview

A diversion system directs water from the surrounding catchment areas to the east and south sides of the facility to South Kemess Creek. The purpose of this system is to divert excess surface water away from the facility while maintaining a storage capacity of 350,000 m³ in the South Diversion Dam impoundment to support fisheries habitat by regulating discharges to South Kemess Creek (~0.05 m³/s) during winter months.

The existing diversion network consists of interconnected components, including the East Diversion Intake (EDI), East, North, and South Diversion Dam Conduits (EDDC, NDDC, SDDC), and the NDD and SDD. The system was originally designed to manage flows from storm events up to the 1-in-200-year, 24-hour storm.

KP completed an assessment of increased tailings and pond loads within the TSF on the existing diversion system to determine whether the conduits could sustain the additional loads from the raised tailings, embankments, and pond while maintaining full functionality. Results indicated that the SDDC and most sections of the EDDC would not be able to support the increased loads (KP, 2025). Consequently, a new diversion system strategy has been developed.

The new diversion system will function similarly to the existing one but will be constructed at a higher elevation to provide gravity flow past the TSF Main Embankment. An East Diversion Dam (EDD) will also be constructed to prevent tailings migration beyond the current facility boundary and the Schedule 2B delineation.

Updated Diversion System

The new diversion system is a gravity-driven network designed to convey flows from the EDD and NDD catchment areas to a pond upstream of SDD where it is routed to the Main Dam spillway, which discharges into Kemess Creek as shown on Drawing C0200. Water collected upstream of the EDD, NDD, and SDD will be conveyed through their respective intake structures into downstream channels or conduits. Three conveyance structures are proposed:

- East Diversion Dam Conduit (EDDC)
- North Diversion Dam Channel (NDDC)
- South Diversion Dam Channel (SDDC).

The diversion channels and conduit have been designed to accommodate the 1-in-200-year, 24-hour storm event for each diversion pond. Inflow rates were determined by calculating runoff from the respective catchment areas and then attenuating it through the pond storage; the resulting flow

represents the design inflow for the diversion system. The dimensions of the channels and conduits for the updated diversion system is presented in Table 18-6.

Table 18-6: Channel/Conduit design parameters

Parameter	Units	EDD	NDD	SDD
Design flow	m^3/s	0.58	1.37	
Design slope (Minimum slope)	%	0.2	0.2	0.2
Total length	m	4,400	550	1800
Intake elevation	masl	1548	1541	1540
Outflow elevation	masl	1540	1540	1537
Pipe diameter	m	1.0		
Channel slope (H:V)			2:1	2:1
Channel base	m		2	2
Channel freeboard	m		0.3	0.3
Flow depth	m		0.5	0.9
Channel depth	m		0.8	1.2

The alignment of all diversion ditches and conduits will follow the updated diversion road layout and will be engineered to maintain the required gravity gradient to ensure proper flow conveyance.

Diversion Dams

The EDD, NDD, and SDD have been designed to accommodate the raise of the facility, provide gravity drainage for the diversion system, and store the 1-in-200-year, 24-hour flood event. The SDD and NDD will be raised to crest elevations of 1,541 masl and 1,544 masl, respectively. In addition, the new EDD will be constructed to an elevation of 1,549 masl to provide flood protection and ensure containment within the facility limits. The construction of the diversion dam raises will be dependent on impoundment tailings and water levels.

The SDD will be raised using downstream construction methods. The raise will incorporate rockfill shell zones on both sides of a central core, as illustrated on Drawing C0211. Filter/transition zone will be incorporated between the central core and shell zones to provide drainage and prevent piping of the low permeability core.

The NDD will be raised from elevation 1,540 masl to 1,544 masl using a centreline raise above the existing dam, as shown on Drawing C0212. This raise will include extending the core and transition zones on both sides, along with upstream and downstream rockfill shells.

The EDD will consist of a zoned earth-fill/rockfill embankment with a low permeability core and with appropriate filter/transitions zones, as shown on Drawing C0212.

Dam Sequencing

The proposed KS TSF construction sequencing is summarized in Table 18-7.

Table 18-7: KS TSF dam raise sequencing

Mine Year	Main Dam (masl)	North Dam (masl)	EDD (masl)	NDD (masl)	SDD (masl)	Main Dam buttress (masl)
4	1512	-	1549	1544	1541	1417
5	1515	-	1549	1544	1541	1417
6	1518	-	1549	1544	1541	1417
7	1520		1549	1544	1541	1417
8	1523		1549	1544	1541	1417
9	1525		1549	1544	1541	1417
10	1527		1549	1544	1541	1417
11	1530		1549	1544	1541	1417
12	1532		1549	1544	1541	1417
13	1534		1549	1544	1541	1417
14	1536		1549	1544	1541	1417
15	1537		1549	1544	1541	1417

Note: (1) Based on filling curve data does not account for sloping beaches. To be refined with sequenced deposition plan. 2. Sequencing elevation required at end of corresponding year.

Materials

The Main Dam will be constructed as a cycloned-sand shell dam, while the diversion and saddle dams will be rockfill-shell structures, all incorporating a low-permeability till core. The rockfill dams will include filter and transition zones, with the diversion dams featuring these zones on both the upstream and downstream sides of the core. Details of all dam construction materials are provided below.

- The core material will consist of locally borrowed glacial till, potentially sourced from Borrow Area 10.
- The filter and transition zones will be developed from processed glaciofluvial deposits and/or NAG rock. The filter and transition materials must meet filter compatibility criteria to prevent piping of the low permeability core.
- The downstream and upstream shell will be constructed using NAG rockfill. Suitable NAG rockfill are anticipated to be sourced from several locations, including the local granular borrow pits, KS NAG Waste Rock Dump, and mining activities.
- Cyclone sand produced through cyclone separation processes. The cellular placement method, consistent with previous construction practices is expected to be an appropriate approach for continued downstream shell development.

KSF Stability Assessment

General

Slope stability analyses were completed to evaluate the stability of the KS TSF embankments at their updated ultimate design elevations. All analyses were performed in SLOPE/W (Seequent, 2024) using the Morgenstern–Price method to determine minimum global and local factors of safety.

The entry–exit slip surface option was used for all dams to identify potential critical failure surfaces. For the Main Dam, additional evaluations were conducted using the block-specified slip surface method to further assess potential failure zones through the GLU and exiting at the base of the Main Dam slope, and at the toe of the buttress.

Design Criteria

The minimum factor of safety required for each loading condition for TSF embankment design have been summarized in Table 18-8. Minimum factors of safety have been adopted from the Health, Safety and Reclamation Code (HSRC, 2024) and the Canadian Dam Association Dam Safety Guidelines (CDA, 2019).

Table 18-8: Minimum factors of safety for dam stability

Loading condition	Minimum factor of safety
Static condition[1]	1.5
Post-Seismic[1]	1.2
Pseudo-Static[2]	1.0

Loading Conditions

The minimum FOS were determined for each of the following scenarios:

- Drained and undrained static loading conditions
- Pseudo Static
- Post-Liquefaction conditions

The facility is classified as "Very High" under the HSRC guidelines (EMLI, 2024), requiring a seismic design 1/2 between the 1/2475 and 1/10,000-year event which corresponds to a PGA of 0.12 g (NRCAN, 2020). A horizontal seismic coefficient of 0.06 g, 50% of the PGA, was applied in the stability assessment.

Model Sections

Typical sections of all the dams are presented in Figure 18-6 to Figure 18-10. This assessment evaluates a typical section of the Main Dam through the valley which was developed based on historic model sections. The SDD section was developed based on the original ground conditions presented in

the initial design (KP, 1997). The NDD, EDD, and North Dam foundation profiles were interpreted using the limited nearby drillhole data and available geological cross-sections. It is noted that variable foundation conditions have been identified beneath the SDD, and these should be further evaluated and confirmed through future studies.

Figure 18-6: KS TSF main dam cross-section

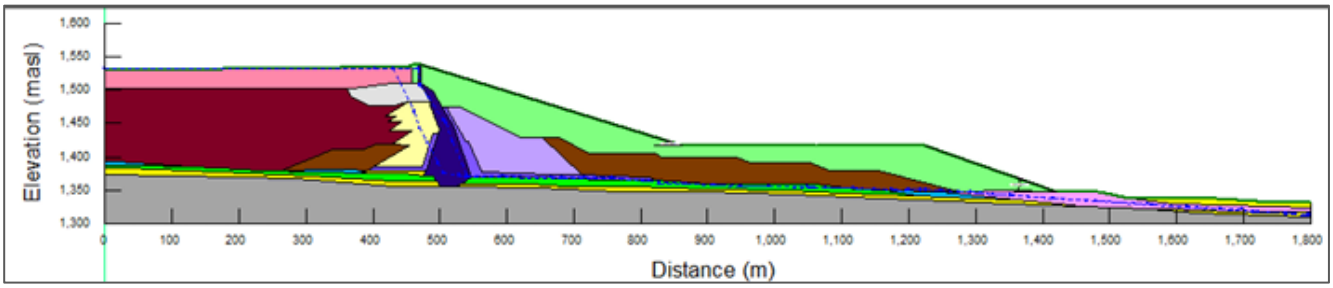
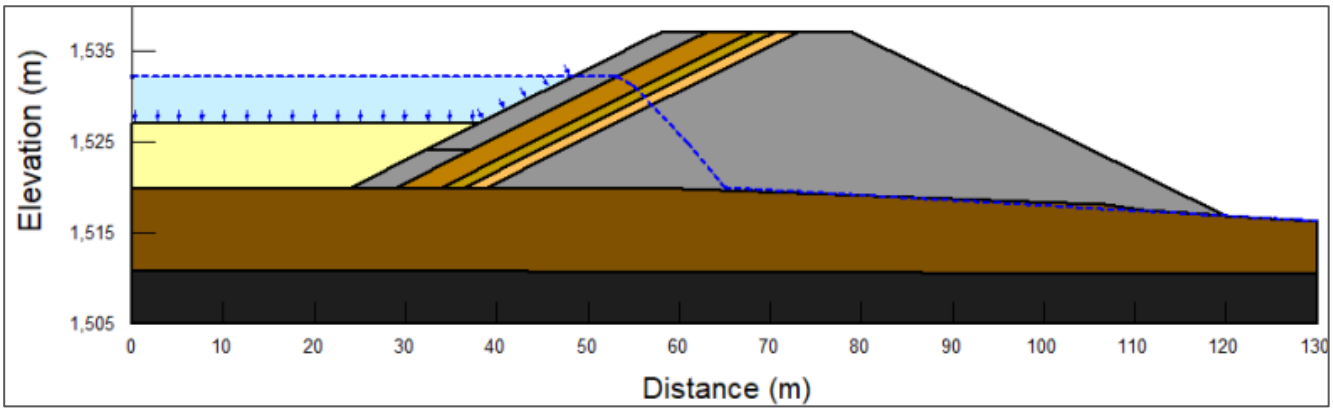
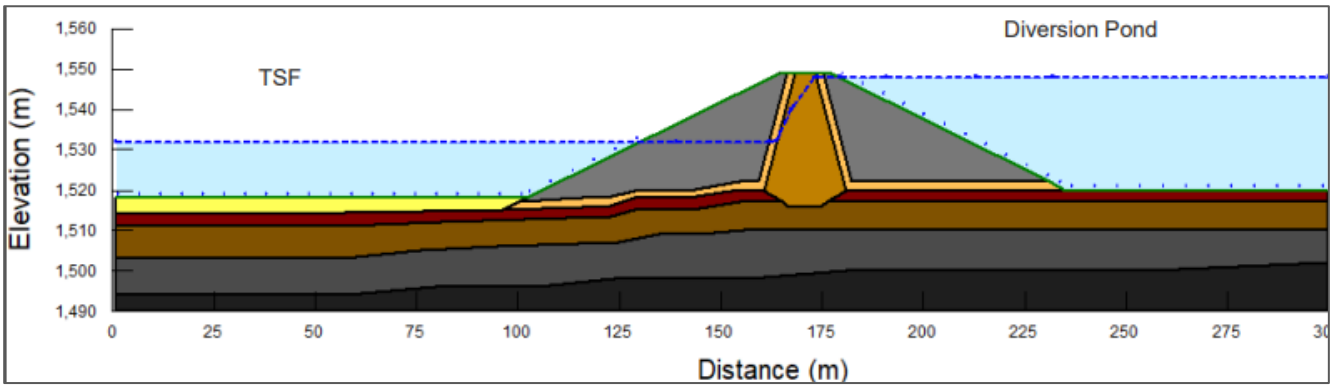


Figure 18-7: North saddle dam cross-section



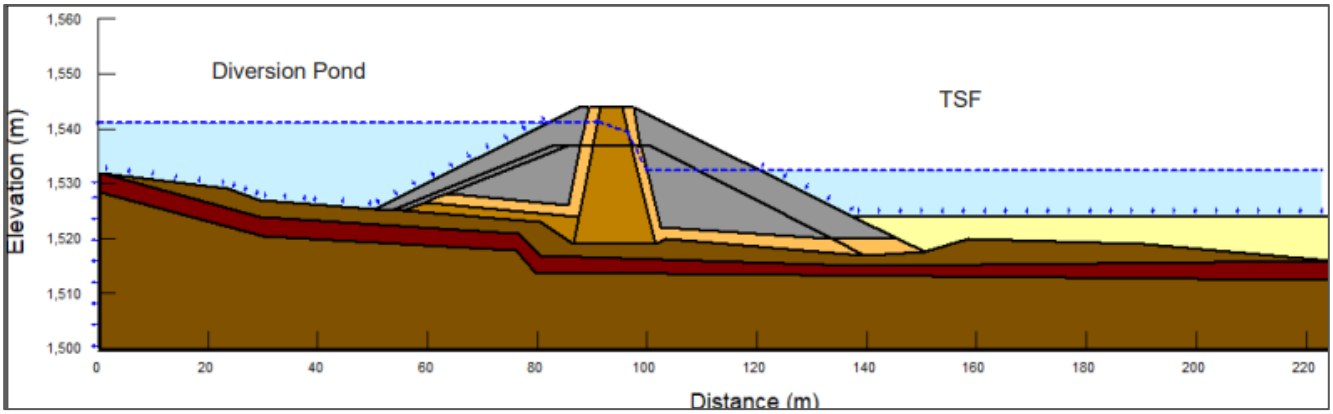
Note: (1) The foundation materials beneath the North Saddle Dam have been interpreted primarily from historical drillholes located at a significant distance, limited site-specific geotechnical data is available for this area

Figure 18-8: East diversion dam cross-section



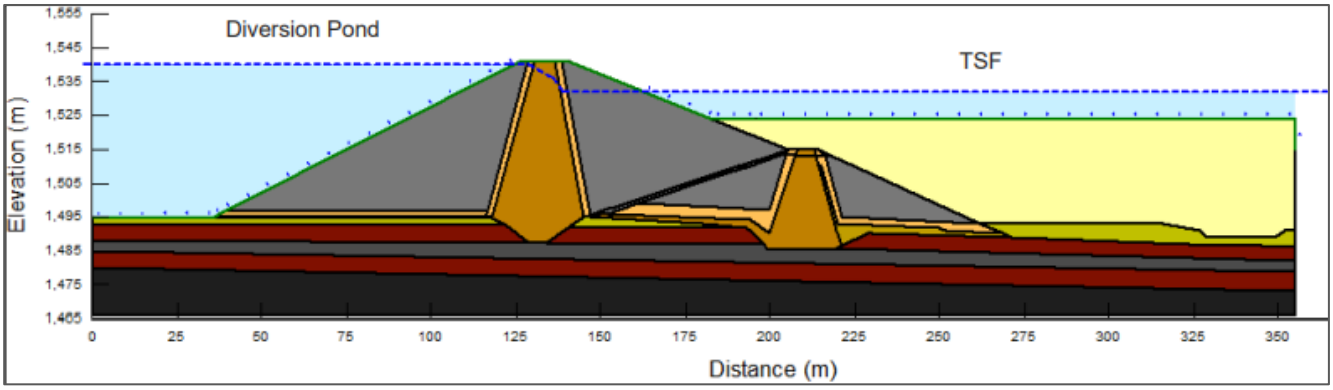
Note: (1) The foundation materials beneath the East Diversion Dam have been interpreted primarily from historical drillholes located along the previously proposed dam alignment.

Figure 18-9: North diversion dam cross-section



Note: (1) The foundation materials beneath the North Diversion Dam have been interpreted primarily from historical drillholes located along the dam alignment.

Figure 18-10: South diversion dam cross-section



Material Parameters

Material unit weights and shear strength parameters used for the slope stability analysis for the Main Dam are based on the parameters presented in the 2023 Dam Safety Inspection stability assessment (WSP, 2024) and are summarized in Table 18-9. The North Saddle Dam and EDD, NDD, and SDD embankment material parameters are based on the embankment material parameters developed in the KUG Feasibility Report (AMEC, 2012).

The GLU exhibits anisotropic strength. This behaviour has been incorporated into the Slope/W analysis using an anisotropic strength function and is represented as the drained case for the GLU using the parameters from WSP 2023 DSI stability assessment (WSP, 2024).

Table 18-9: Material strength parameters used in stability analysis

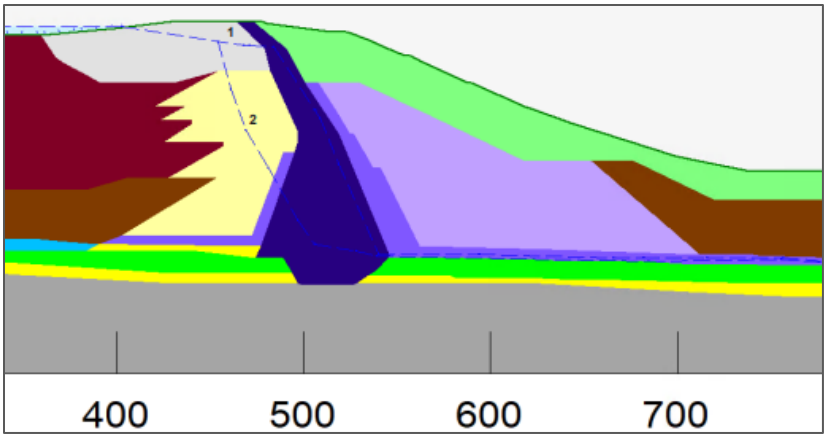
	Material	Unit weight (kN/m)	Friction angle (°)
Main Dam Embankment Fill	Zone B1 (transition filter)	19.0	35
	Zone C – DS (waste rock)	23.0	37
	Zone C – US (waste rock)	23.0	45
	Zone E (buttress random fill)	21.0	36
	Zone S (till core)	21.5	36
	Compacted cyclone sand	17.3	34
	Cyclone sand (upstream)	17.3	27
	Historic tailings[5]	21.0	
	Proposed tailings[5]	19.0	27
Saddle and Diversion Dam Fill	Filter	21.5	26
	Transition	21.5	36
	NAG rock fill	23.0	36
	Core (till)	21.5	36
Foundation Materials	Glaciofluvial	21.5	36
	Colluvium	21.5	36
	Overburden	22.0	37
	Ablation till	21.5	36
	Glaciolacustrine deposit[1]	19.5	22 (oblique slip surface) 10.5 (horizontal slip surface)
	Till	21.5	36
	Weathered bedrock	21.0	36
	Bedrock	Impenetrable	

Notes: (1) A peak undrained strength ratio of 0.185 (minimum strength 45 kPa) was applied for the GLU in the undrained model scenarios. (WSP, 2024) (2) Main Dam materials parameters based on the parameters presented in WSP 2023 DSI stability assessment. (WSP, 2024) (3) Diversion and North Saddle Dam parameters based on parameters presented in the KP 1997 Final Design Report stability assessment for the SDD (KP, 1997). (4) Proposed tailings material parameters from KUG TSF Feasibility Report (AMEC, 2012). (5) Cohesion assumed to be 0 kPa for all materials. (6) Residual undrained strength parameters of 0.05 was applied to the tailings in the post-seismic case. (WSP, 2024)

Pore Pressure Conditions

Two piezometric lines were defined for the Main Dam stability analyses; one piezometric line has been applied within the embankment fill, and the other to the foundation materials based on the observed 2023 piezometric water levels presented in the 2023 DSI (WSP, 2024). The piezometric line within the embankment fill represents seepage from the pond elevation through the central core and downstream zones. This is applied to all dam materials, including the tailings. The piezometric line in the foundation represents seepage from the pond through the upstream shell, the coarse filter/drainage blanket, and then downstream through the foundation (Figure 18-11). This line is applied to all foundation materials.

Figure 18-11: Simplified section for analysis



The phreatic surfaces used for the EDD, NDD, SDD, and North Dam stability analyses were based on the maximum normal operating TSF pond elevation of 1,532 masl on the upstream side of each dam. For the downstream sides of the diversion dams, phreatic surfaces were assigned using the operating water levels within the diversion ponds: 1,548 masl for the EDD, 1,541 masl for the NDD, and 1,540 masl for the SDD.

An additional phreatic surface scenarios was evaluated for early-operation conditions. In this case, the TSF pond is at a minimum elevation due to limited initial tailings deposition, resulting in a lower TSF phreatic surface while diversion channel levels remain at their normal operating elevations.

In all models, the seepage path through the dams was defined to reflect minimal head loss across the core, with drainage occurring primarily through the filter and transition zones in the direction of head loss.

Stability Analysis Results

The results of the stability analyses, summarized in Table 18-10, Table 18-11 and Table 18-12, indicate the TSF embankments achieve the required FOS.

Table 18-10: KS TSF stability analyses results

Cases	Slip surface method	Drained strength		Undrained strength	
Failure plane		Main Dam	Buttress toe	Main Dam	Buttress toe
Static	Entry Exit	1.54		1.64	
	Block Specified	1.53	1.60	1.54	1.63
Pseudo-static	Entry Exit	1.26		1.26	1.22
	Block Specified	1.22	1.18	1.22	1.19
Post Liquefaction	Entry Exit	1.43		1.42	
	Block Specified	1.39	1.48	1.38	1.46

Table 18-11: Localized failure Stability Analyses results

Cases	Slip surface method	Drained strength		Undrained strength	
Failure plane		Main Dam	Buttress	Main Dam	Buttress
Static	Entry Exit	2.21	1.96	2.25	1.96
Pseudo Static		1.82	1.64	1.82	1.64
Post Liquefaction		2.06	1.96	2.06	1.96

Table 18-12: Diversion dams and North dam Stability Analyses results

Cases	EDD		NDD		SDD		North Dam
Phreatic surface conditions	Maximum operating levels	Maximum diversion Minimum TSF WL	Maximum operating levels	Maximum diversion Minimum TSF WL	Maximum operating levels	Maximum diversion Minimum TSF WL	Maximum operating TSF WL
Static	1.50	1.61	1.54	1.58	1.99	1.92	1.90
Pseudo Static	1.26	1.34	1.30	1.37	1.63	1.60	1.60
Post Liquefaction	1.50	-	1.54	-	1.98	-	1.88

Roads and Additional Infrastructure

The road along the south side of the KS TSF, which currently provides access to the East Intake and proposed EDD, will be inundated by tailings and must be relocated to maintain access. Relocation will require excavation through a steeply sloping area, including sections with 2H:1V side slopes. The reconstructed diversion road is assumed to be 6 m wide to accommodate two-way traffic, with an additional 4 m allocated for the EDDC pipe deck and a safety berm.

Access to the North Dam will also require the construction of a new excavated road, designed to the same specifications as the relocated diversion road. In addition, the upper portion of the spillway and the adjacent roadway will be impacted by the embankment raises and will require removal and reconstruction in a new alignment.

Closure Design

Closure of the KS TSF will include reinstating a spillway channel on the left abutment of the Main Dam, tying into the existing, unaffected spillway. The spillway invert will be set to maintain a water cover over the tailings to an elevation of 1,533 masl. The spillway will be sized to convey the PMF, including PMP and snowmelt contributions, resulting in an inflow of 133 m^3/s.

The spillway will have a base width of 25 m, a minimum depth of 4 m, side slopes of 2H:1V, and an approximate length of 1 km. The TSF embankments and exposed rougher tailings surfaces will be covered with overburden and vegetated to blend with the surrounding landscape.

TSF Closure Plans

Preliminary closure plans for the KS TSFs include placement of a vegetated cover over the surface and downstream embankment slopes to promote long-term stability and rehabilitation to blend in with the

surrounding area. The KUG TSF impoundment is anticipated to require a water cover to mitigate oxidation of cleaner tailings. The closure designs for both facilities include construction of a closure spillway for passive water management in perpetuity. The requirement for water treatment at closure should be completed as part of future studies.

18.5 TAILINGS PIPELINE

18.5.1 Startup to Year 3

At mill startup until Year 3, the plant tailings will flow by gravity in a pipeline from the mill to an existing mixing box in pump house 1. Three of the existing five pumps will be refurbished and a new tailings pipeline will transport whole tailings to the KUG TSF.

18.5.2 Operating Year 4 for Remaining LOM

Starting in Year 4, the tailings deposition strategy will change as the leach plant will becomes starts processing the cleaner scavenger tailings. The high-pyrite tailings from the leach circuit will be pumped directly into the KUG TSF for the remaining mine life.

Rougher flotation tailings will flow from the process plant to pump house 1 to be pumped to a cyclone sands plant that will discharge into the Kemess South TSF. To reach the TSF, two existing 26-inch tailings pipelines (one operational and one on standby), and all the necessary pumps, will be replaced or refurbished depending on equipment condition.

18.6 WATER MANAGEMENT

18.6.1 Reclaim Water

Startup to Year 3

The reclaim water supply for mill operations will initially be sourced from the KUG TSF, with the systems designed to maximize reuse of existing infrastructure while incorporating selective upgrades.

At the KUG TSF, the existing reclaim barge and associated pumps will be refurbished to restore operational reliability. New discharge piping will be routed from the barge to connect with the existing on-shore header. From this point, reclaim water will flow through the existing twin pipelines to the existing reclaim water transfer station. Piping infrastructure will be refurbished as required to reinstate the systems.

Operating Year 4 for Remaining LOM

Starting in Year 4 of operations the reclaim water configuration will change to account for the change in tailings deposition into both the KUG TSF and the KS TSF. A reclaim water barge will be relocated to

the KS TSF to re-establish reclaim pumping there. A new 7 km buried HDPE pipeline will be installed to transfer reclaimed water from the KS TSF to the existing reclaim transfer station, where it will join the reclaim water supply from the KUG TSF. Existing piping infrastructure will be refurbished as required to reinstate the systems.

18.6.2 Kemess South Waste Rock Storage Facility Contact Water

The contact water management system developed and maintained during the care and maintenance period will remain in operation throughout the life of mine. Runoff and seepage generated from the waste stockpile will be collected and conveyed to the Water Treatment Plant (WTP) for treatment prior to discharge.

18.6.3 Kemess North Waste Rock Storage Facility Contact Water

For the WRSF, contact and seepage water will be collected in dedicated collection ponds and pumped into the existing site water management system for treatment and discharge. Further engineering and hydrological assessments to refine the collection and conveyance systems are planned in subsequent phases of study.

18.6.4 Water Treatment Facility

There is an existing water treatment plant located at the Kemess South plant site. The plant is comprised of two circuits and designed for the LOM treatment of water:

- A metals removal circuit which includes precipitation, filtration and removing metal contaminants at a feed flow rate of 170 L/s.
- A selenium removal ion exchange circuit (Selen-IX™), removing selenium at a feed flow rate of 65 L/s.

The plant will be operational throughout the project to treat water as a method to regulate selenium and metals levels. The site water balance is in closed circuit and no water discharge to the environment is planned or expected for roughly the first five years of operations.

19 MARKET STUDIES AND CONTRACTS

19.1 MARKETABILITY

Upon achieving a commercial production stage, the Kemess Mine will produce gold and silver doré, and a copper and gold concentrate. Most gold and silver are recovered into the copper concentrate, with the remainder extracted through the leaching circuit and subsequently cast into doré bars. . The doré bars will be shipped to third-party refiners for final refining and sale. Based on common industry practices, payable metal content from doré sales is estimated to be approximately 99.8% for gold and 90% for silver, relative to the contained metal in the doré.

Copper concentrate produced through a conventional flotation process will be sold and shipped to smelters. Copper grade of the concentrate is projected to be around 21%, which is expected to result in a payable factor for copper of about 94.8%, making the concentrate from Kemess Mine readily marketable.

Gold, silver, and copper are traded in mature global markets with reputable refiners and smelters located throughout the world. The long-term outlook for gold remains strong, supported by sustained central bank buying, safe-haven demand, and macroeconomic uncertainty, with current gold prices at historically high levels. Silver demand is projected to remain strong due to its critical role in the solar, battery-electric vehicle, and electronics sectors, supporting a bullish long-term price trend. Copper demand is expected to remain strong over the long term, driven by electrification, renewable energy, and infrastructure growth, supporting favourable pricing and marketability for concentrates. The entirety of silver production from the Kemess Mine will be sold to Triple Flag through the Kemess silver stream (“Kemess Silver Stream” - more details are provided below).

19.2 METAL SALES

Gold and silver doré will be sold to the London Bullion Market Association (LBMA) accredited refiners with extensive experience in refining gold and silver doré produced by Canadian mining operations. The Kemess site is equipped with an airstrip that supports charter flights for personnel and light cargo, and gold and silver doré will be transported using air logistics.

Copper concentrate from the Kemess Mine will be sold to established smelters in Asia, leveraging the experience of Centerra Gold in selling similar product to smelters in South Korea, China, and Japan from its Mount Milligan operation in British Columbia since 2016. The concentrate will be trucked approximately 380 km via the mine access road to MacKenzie. Centerra maintains a railhead loadout facility at MacKenzie that is currently used at full capacity for Mount Milligan Mine concentrate shipments. Centerra previously purchased a plot of land near the existing rail loadout facility and plans to construct a new facility to accommodate the Kemess concentrate production. From MacKenzie, the

concentrate will be transported by rail to British Columbia ports and shipped via ocean freight to overseas smelters.

19.3 KEMESS SILVER STREAM

On May 17, 2018, Centerra announced that its wholly owned subsidiary, AuRico, entered into agreements with Triple Flag Mining Finance Bermuda Ltd (“Triple Flag”) in connection with the sale of AuRico’s royalty portfolio and a silver stream on the Kemess Project (“Kemess Silver Stream”) for combined aggregate proceeds of $200 million. The Kemess Silver Stream consists of the sale of 100% of the silver production from the Kemess Mine. Triple Flag agreed to pay AMI cash consideration of $45 million as an advance payment, payable in tranches of $10 million, $10 million, $12.5 million and $12.5 million, which become due upon the occurrence of certain milestones following a construction decision with respect to the Kemess Project and on the three succeeding anniversaries of that date, respectively. Under the terms of the Kemess Silver Stream, Triple Flag will receive 100% of the silver production from the Project and will make ongoing payments of 10% of the then current market price for each ounce of silver delivered.

After incorporating the impact of the silver stream arrangement, total Project cash proceeds from silver sales and the silver stream advance payments are estimated to represent approximately 0.5% of total Project revenue. The full impact of the silver stream arrangement is incorporated into the economic analysis presented in this PEA.

19.4 CONTRACTS MATERIAL TO THE PROJECT

Other than the Kemess Silver Stream, currently there are no material sales contracts in place for the Project. Centerra Gold anticipates that any future doré sales agreements will reflect standard industry practices and be consistent with terms commonly applied to doré supply contracts globally. Similarly, terms for copper concentrate sales are expected to align with standard industry practices and be comparable to those currently in effect for copper and gold concentrate sales from the Mount Milligan Mine.

Aside from product sales agreements, the largest contracts during the LOM are expected to cover the supply of major items such as mine equipment, mine and processing consumables, electric power, procurement of bulk commodities, third-party underground mining services, camp management, and flights logistics. These contracts will be negotiated and renewed as required prior to and throughout the LOM.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Kemess Mine is located in a mountainous region of north-central British Columbia and is currently in a state of care and maintenance. Kemess mine possesses both provincial and federal Environmental Assessment (EA) authorizations, an EA Certificate from the BC Environmental Assessment Office (EAO) and a Federal Decision Statement (DS) from the Impact Assessment Agency of Canada. The mine also possesses a Fishers Compensation Agreement (FCA) from the federal Department of Fisheries and Oceans (DFO) and provincial Mines Act (MA) and Environmental Management Act (EMA) permits from the BC Ministries of Environment and Parks and BC Ministry of Mines and Critical Minerals (MCM). Finally, Kemess Mine also possesses ancillary permits from the BC Ministry of Water, Lands and Resource Stewardship (WLRS).

All the permits have been kept current and compliance against them reviewed to ensure that the mine site is in a favourable state of operational readiness to support re-start ramp-up and construction activities. A streamlined permitting process will be pursued to complete necessary amendments to existing permits and authorizations to accommodate the project revisions described in this PEA.

Comprehensive environmental studies are being conducted to support the Kemess Restart Plan (KRP), which proposes a combination of open pit and underground mining in the Kemess North or ‘Main Zone’ area. The KRP represents a strategic shift from the previously permitted block cave mining method to a combined open pit and longhole underground operation. This transition necessitates updates to the environmental baseline and predictive models of the project, leveraging data collected since 2002 while addressing specific gaps related to the expanded footprint in the Main Zone mining Area.

Engineering, construction, operation and management of mine facilities and components utilize criteria for responsible management to meet regulatory obligations. Environmental management plans and internal compliance reviews and audits guide the compliance and monitoring programs at the mine. The mine continues to engage with indigenous nations, local communities, and stakeholders to share information and collaborate on environmental and/or social development initiatives.

20.1 ENVIRONMENTAL STUDIES

The regulatory framework for mining in British Columbia and Canada provides rigorous processes for assessing the Kemess Project and its potential environmental and socio-economic impacts.

The 2017 Kemess Underground Project EA and permit applications included an extensive list of environmental baseline studies and effects assessments. These included comprehensive studies and assessments related to terrain and soils, air quality, noise, water quality and quantity, vegetation, wildlife, fisheries and aquatic resources including benthic invertebrates, periphyton, sediment quality,

and habitat. Studies also included visual and aesthetic resources, land use, archaeology and heritage resources, social and economics, and human health. Potential environmental and socio-economic effects and mitigations were fully considered in the environmental assessment and mine permitting processes. Approvals were received based on the mitigations and management plans within the applications. Previous baseline collection programs and data from ongoing monitoring programs will be utilized in the assessment of new areas, as available.

New environmental studies are required to support the environmental assessment for permitting the KRP. ERM consultants were retained to complete a Permitting Gap Analysis in 2025 to identify any environmental areas or disciplines where additional studies may be required to meet or exceed provincial and federal permitting requirements. The results of this Gap Analysis are now available and Centerra is using the knowledge to scope new or modify ongoing studies planned for 2026.

These studies include, but are not limited to, water quality and quantity, fisheries and aquatic resources, archaeology, soils, vegetation, and wildlife. The studies will assess new areas to be disturbed for new mine waste storage facilities, the conversion of the existing permitted block cave subsidence zone to an open pit configuration, new conveyor alignment, etc. Field work to complete the environmental studies commenced in 2024. with the aim to have baseline collection completed by the end of 2027/early 2028.

20.1.1 Surface Water and Water Quality

In 2025, ERM updated the site-wide water balance and quality models using GoldSim™ software to simulate current care and maintenance conditions, 18 years of active operations, and long-term closure. Modeling results confirm that all contact water from the Main Zone Area—including the Main Zone Pit, Nugget Pit, and the PAG WRSF—will be captured and routed south to the existing treatment infrastructure, ensuring no impacts on Amazay Lake. The water balance model predicts that annual contact water volumes will rise from 1.1 Mm^3 in Year 1 to a peak of approximately 2.5 Mm^3 by Year 13.

The primary parameters of concern for the project include copper, sulphate, fluoride, and cadmium. Predicted concentrations of total copper and dissolved cadmium in the Main Zone Area are expected to exceed current permit limits during operations, requiring the water to be treated through the existing Water Treatment Plant (WTP) in the south area. Although not a requirement for the mine restart, a Site Performance Objective (SPO) is currently being developed to establish long-term selenium permit limits for Waste Rock Creek, supported by ongoing studies such as the Bird and Amphibian Selenium Bioaccumulation study initiated in early 2025 to manage selenium inputs from a previously operated WRSF in the Kemess South Mine Area.

20.1.2 Hydrogeology and Groundwater

The groundwater numerical model was updated in 2025 using FEFLOW to simulate the effects of the Open Pit and underground mine development. The model domain is highly detailed, featuring 16 vertical layers and approximately 2.8 million nodes to maintain stability across the region's steep topography. Modeling indicated that drainage in the underground mine area would result in a water table drawdown of approximately 250 m compared to initial baseline conditions. Simulations were conducted for multiple scenarios, including Mine Year 9, Mine Year 18, and post-closure, incorporating various climate scenarios (wet, dry, and average) to assess the sensitivity of the groundwater system. The model predicts that the Main Zone Pit will remain a net groundwater sink even after flooding to the spill elevation of 1,571 masl, which effectively reduces contact seepage reporting to East Cirque Creek. Sensitivity simulations were also conducted for 100-year wet and dry climate scenarios to assess long-term water balance stability.

20.1.3 Geochemistry and Metal Leaching

The KRP involves the excavation of approximately 117–164 Mt of waste rock, of which a dominant portion (158.5 Mt) is categorized as PAG. Based on a Net Potential Ratio (NPR) threshold of ≤ 2.0, the Takla-Gossan (median NPR 0.0084), Takla Group (median NPR 0.041), and Toodoggone Formation (median NPR 1.9) rock units have all been identified as PAG. The Takla Group, which comprises the majority of the mine rock, is particularly susceptible to weathering because it lacks significant acid-buffering carbonate minerals like calcite and dolomite.

Environmental consultants Lorax conducted a geochemical gap analysis in 2024, noting that while the historical static database is robust, additional kinetic testing is required for materials from the 2024 exploration program to refine source term predictions. This is because many previous kinetic samples were collected from the block cave subsidence zone and are located outside the current proposed Main Zone and Nugget pit footprints. To ensure precise waste management, an algorithm using calcium/sulphur (Ca/S) ratios as a proxy has been developed for the block model to delineate PAG, potentially non-acid generating (PNAG), and non-acid generating (NAG) material.

Current source terms indicate that seepage from the Main Zone Pit will initially be influenced by the Leach Zone and Broken Zone material, which produces acidic runoff due to historic oxidation and high existing sulphate levels. As mining progresses, seepage is expected to transition to unoxidized Takla material, which remains neutral for longer durations. Kinetic testing of the Takla Group shows neutralization potential (NP) depletion times ranging from 0.3 to 100 years. Approximately 68% of the Takla material is expected to become acidic within a 20-year timeframe, while the Leach Zone material is immediately acid-generating, with an onset time of less than one year.

To account for field conditions, a reactive mass scaling factor of 20% was applied to laboratory results for estimating loading rates from the PAG WRSF and pit backfill, representing the weight percentage of blasted rock within the reactive size fraction (<6.3 mm). For pit walls, this factor was refined to 20% for the blast damaged zone and 5% for the blast fractured zone. Additionally, a flushing factor of 20% was assumed for stockpiles to account for preferential flow paths that bypass reactive surfaces. Metal leaching of copper, selenium, cadmium, and sulphate is expected; however, geochemical studies show that the Kemess Main Zone area has significantly lower selenium content than the historical Kemess South mine.

20.1.4 Fish and Aquatic Resources

The Kemess site possesses an extensive biological database for fisheries and aquatic resources, with continuous data collection dating back to 1992 for sediment quality and fisheries resources. This long-term monitoring satisfies commitments under the Fisheries Compensation Agreement and the Fish and Aquatic Effects Monitoring Plan (FAEMP).

Bull Trout and Dolly Varden Populations

The Bull Trout is the primary species of conservation concern in the Thutade Lake watershed, while the Dolly Varden trout population is monitored due to historical habitat losses related to the original mine construction.

- Adult abundance: In 2024, the Kemess Creek watershed recorded 230 Bull Trout redds, the highest count since monitoring began in 1994. This represents a significant rebound and the fourth consecutive year of high redd counts, suggesting the population is currently self-sustaining and robust.
- Juvenile densities: After a period of concern regarding low recruitment, Bull Trout parr densities (age 1+ and older) rebounded between 2021 and 2024, meeting or exceeding established monitoring thresholds for three consecutive years. While char fry (young-of-year) densities were below historical norms in 2024, they remained within acceptable monitoring targets.
- Life history and growth: Synoptic analysis of fin ray samples suggests that improved growth and survival in the adult rearing environment of Thutade Lake may be driving the recent increases in adult abundance. Additionally, warmer winter water temperatures in South Kemess Creek since 2013 (increasing from 1.8°C to 3.6°C) may be contributing to larger char fry sizes due to earlier emergence and a longer growing season.

Water Quality and Aquatic Health

Monitoring programs evaluate the impact of permitted discharges from the Kemess Underground (KUG) diffuser into Attichika Creek and from the Kemess South (KS) TSF spillway.

- Toxicity testing: In 2024, acute toxicity monitoring indicated a 100% survival rate for rainbow trout and Daphnia magna at discharge and receiving environment stations, including the KUG initial dilution zone.
- Periphyton and benthos: While overall environmental quality remains stable, a shift in the periphyton community was observed at the near-field station ATT-IDZ, where mine discharges may be selecting for certain diatom taxa, resulting in a reduction of taxonomic richness from 21 to 11 taxa. Benthic invertebrate communities in South Kemess and lower Kemess creeks have not been substantially altered by mine effluent, with differences in community composition primarily driven by habitat factors such as the presence of D. *geminata mats*.
- Metals in biota: Selenium and mercury concentrations in Slimy Sculpin and Bull Trout tissue were greater than BC guidelines for the protection of aquatic life at both reference and exposure areas. These elevated levels are considered to be driven by natural background mineralization rather than mine activities.

Mitigation and Habitat Offsetting

Active management continues to maintain the functionality of historical fish habitat compensation projects:

- Side channel maintenance: The constructed juvenile rearing side channel (TP-5) has experienced fine sediment build-up and compaction. In September 2024, site staff modified intake flows to facilitate channel flushing during the 2025 freshet to restore habitat suitability.
- Migration passability: Maintenance was conducted at the El Condor Creek culvert in 2024, where cinder blocks were installed to provide velocity breaks and restore juvenile fish passage to Kemess Lake.
- Selenium management: Current compliance with Permit PE 15335 is maintained through natural attenuation in the pit water body and improved seepage collection. A one-time Bird and Amphibian Selenium Bioaccumulation Study was initiated in 2025 to support the development of a Specific Performance Objective (SPO) for long-term selenium limits.

20.2 TAILINGS AND WASTE ROCK DEPOSITION MANAGEMENT, MONITORING AND WATER MANAGEMENT

20.2.1 Tailings Management

There are two existing and permitted Tailings Facilities, Kemess South TSF and Kemess Underground TSF (previously operated Kemess South Pit) that will be utilized for the KRP. The Kemess TSFs are designed by professional engineers and constructed, operated, and monitored on the advice of an external Engineer of Record (EoR). Centerra has implemented a five-step framework in accordance with the Canadian Dam Association Dam Safety Guidelines. The process involves routine monitoring, staff inspections, annual Engineer of Record inspections, Independent Tailings Review Boards (ITRB), and Independent Third-Party Dam Safety Reports. Each TSF is managed to maintain structural integrity and ensure worker, environmental and public safety. In addition, TSF operation is informed by, and routinely checked against, guidance from the Canadian Dam Association, Mining Association of Canada, and the International Commission on large dams.

The design, construction, operation, maintenance, and surveillance of the TSF's involves a multidisciplinary team of professionals. Oversight of the tailings management team includes the Mine Manager and executive governance personnel of Centerra. The team includes the engineering, environmental, operations, and maintenance departments of Kemess Project, the EoR, design engineers and external professionals as appropriate. Risks are managed through a process of identification, assessment of practicable solutions, implementation of change, and observation. The team works together to achieve the fundamental objective of continuous improvement and safe management of the TSF and associated water management structures.

Tailings Storage Strategy

Tailings management for the Kemess restart leverages existing infrastructure to minimize new surface disturbance while providing secure long-term storage for approximately 261 million tonnes (Mt) of tailings over a 17-year period. The strategy is executed in two primary phases:

Phase 1: Kemess Underground (KUG) TSF (Years 1–4)

Starting with the resumption of ore processing in Year 1 (2031) of the Project, bulk tailings will be directed to the existing Kemess South (KS) Open Pit, which is being repurposed as the KUG TSF.

- Dewatering: To prepare for deposition, the KUG TSF is currently being dewatered at a maximum target rate of 10.3 Mm³ per year (approximately 3,420 m³/h or 950 L/s). To prepare for deposition when ore processing begins in 2031. As of late 2025, the water elevation in the KUG TSF is above 1,220 masl. According to the current modeling results, the water level is projected to be drawn down to a minimum of approximately 1,160 masl by 2031. This

drawdown is necessary to provide sufficient storage capacity for “whole tails” (bulk tailings) during the first three years of ore processing (Years 1–3). The required dewatering is expected to be achieved by 2029, which is well in advance of the late 2031 production target.

- Storage capacity: The facility will provide approximately three years (35 Mt) of bulk tailings storage (Years 1 through 3). Unlike previous plans, the current restart strategy does not involve an expansion of this facility; it will be used at its current capacity with a spillway elevation of 1,255 masl.

Phase 2: Kemess South (KS) TSF Expansion (Years 4+)

Starting in Year 4, bulk “rougher” (de-sulphured) tailings will transition to the existing KS TSF.

- Facility expansion: The KS TSF will undergo a 26-meter dam raise, increasing the crest elevation from approximately 1,511 to 1,537 masl. This expansion will provide approximately 230 Mt of additional storage capacity.
- Construction method: The raise will be executed in stages using a two-stage cyclone sand system, returning to the method used during the operation of the original mine. Approximately 10.3 Mm^3 of non-acid generating (NAG) cyclone sand underflow will be used for dam and buttress construction.
- Ancillary Structures: To support the increased dam elevation, the project will construct several smaller structures, including an East Diversion Dam, a relocated South Diversion Dam, and a raise of the North Diversion Dam to manage the site water diversion system.

Subaqueous and Specialty Tailings Management

A critical component of the environmental strategy is the permanent isolation of reactive and chemically treated materials.

- Cleaner scavenger tailings: Throughout the life of the mine, “cleaner” tailings—which contain high pyrite levels and residual cyanide from the proposed gold leach circuit—will be stored sub-aqueously in the KUG TSF. This prevents acid generation and isolates chemical reagents from the broader environment.
- Paste backfill: For the Kemess restart, longhole underground mining requires paste backfill for structural support, utilizing approximately 31 Mt of rougher tailings between Year 3 and LOM. To de-risk the execution and reduce capital costs, the underground mining rate was optimized from 12,000 tpd to 8,000 tpd, allowing the project to utilize a single-module backfill plant instead of two. Located near the underground portal, this plant has a capacity of 2,880 m^3/day (at 60% utilization) and will produce paste with an estimated strength of 400 kPa using an average binder addition of 5.2% by mass. This strategy provides a secondary

environmental benefit by reducing the total solids deposition required in surface TSFs, which increases available water storage capacity and mitigates long-term closure risks

Potentially Acid Generating (PAG) Waste Rock Storage Facility (WRSF)

The management of 158.5 Mt of PAG generated by the Kemess Main Zone open pit mine is one of the central design challenges for the Kemess Restart Project.

The proposed PAG Stockpile will be a sub-aerial dry-stack facility located in the East Cirque Valley, northeast of the proposed Kemess Main Zone pit. The facility is designed to provide secure and permanent storage for approximately 158.5 Mt (72 Mm^3) of potentially acid-generating waste rock over the LOM. The stockpile will reach an ultimate elevation of 1,713 masl with overall 2H:1V planar side slopes.

Foundation preparation consists of the stripping of topsoil, organic materials, and unsuitable overburden followed by basin shaping for underdrain installation. To prevent contact water from infiltrating the underlying environment, the entire 764,856 m^2 footprint will be lined with a low-permeability glacial till basal liner.

The WRSF features a sophisticated dual-layer water management system designed to capture seepage and meteoric water. The Foundation Collection System (FCS), located above the liner, is anticipated to collect the majority of meteoric water percolating through the waste rock. An Underdrain Collection System (UCS) will be installed beneath the liner as a contingency system. Both systems direct seepage through a network of perforated pipes toward the North Collection Pond (NCP). The NCP is to be excavated into original ground, lined with glacial till to limit seepage losses, and will include a pump station to transfer collected water to a treatment facility. The pond is designed to contain a 1-in-200-year, 24-hour storm event, with a design volume of 80,000 m^3. The water management plan incorporates industry best practices for managing by utilizing a robust containment and collection system designed to protect groundwater and surface water. By lining these ponds with glacial till to limit seepage losses and routing captured water to a treatment plant prior to discharge, the design ensures environmental compliance in accordance with the Health, Safety and Reclamation Code for Mines in British Columbia.

To manage non-contact water, four diversion channels will be constructed adjacent to the facility, in compliance with permit conditions. These ditches are initially sized for a 1-in-10-year storm but will be upgraded at closure to convey a 1-in-200-year storm event. Non-contact water is routed either to the East Cirque Valley watershed or to a South Diversion Pond (SDP), which has a design volume of 100,000 m^3.

The facility has been assigned a Moderate hazard classification according to the Health, Safety and Reclamation Code for Mines in BC.

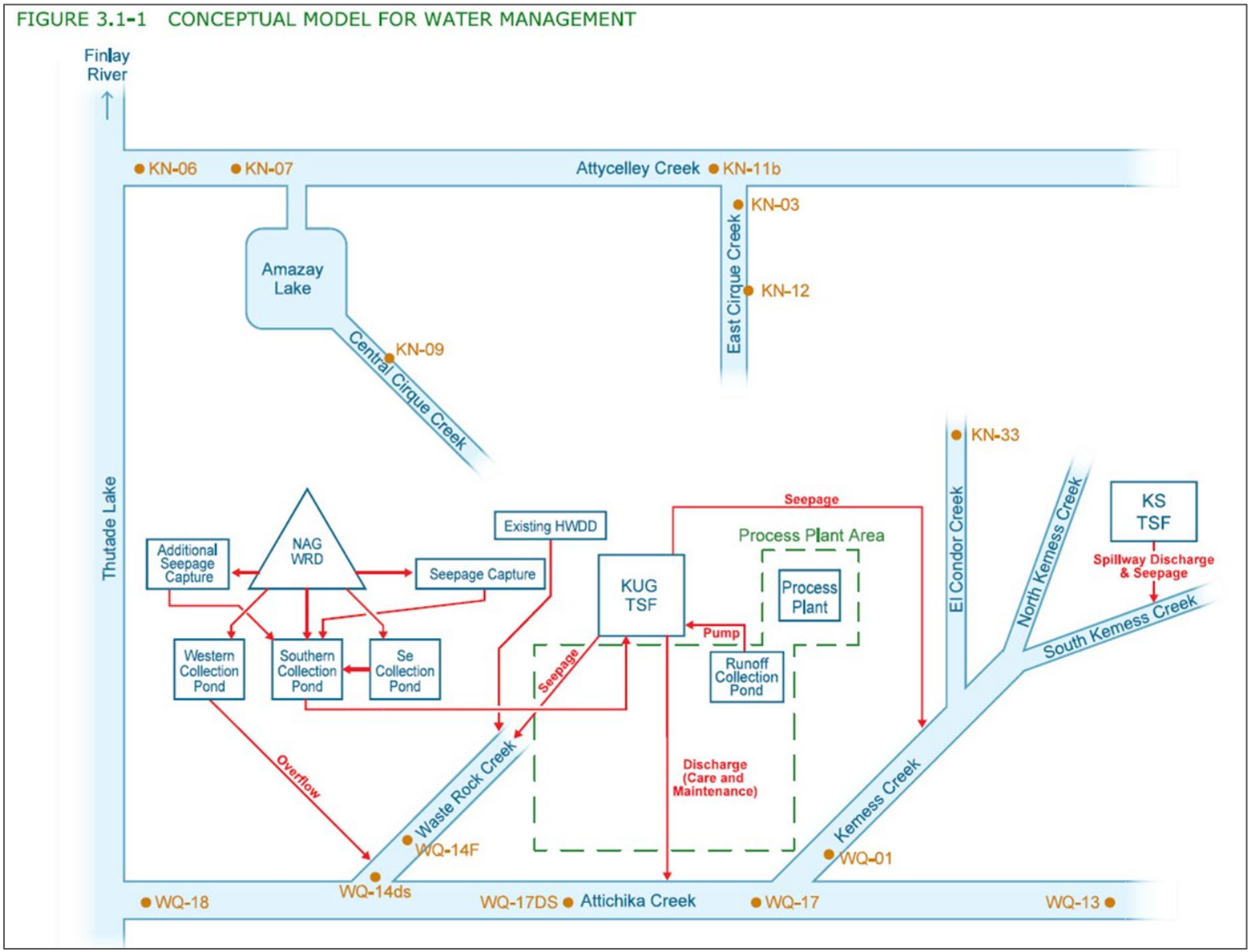
Conceptual closure plans involve regrading slopes to a planar 2H:1V configuration and installing a multi-layer cover. This cover will include approximately 1.0 m of earth-fill topped with 0.3 m of growth medium to limit water infiltration and oxygen ingress. The surface will be re-vegetated to match the surrounding environment, supporting the long-term goal of passive water management in perpetuity.

Water Management

Water management for the Kemess restart focuses on isolating contact water from the North Main Zone Mining Area to ensure no impact on the Amazay Lake watershed. The strategy involves distinct collection and treatment systems for the waste rock storage facility near the pit and the two tailings storage facilities near the plant.

The water balance is managed by recycling contact water for mill processing, which reduces free water inventory and mitigates downstream water quality risks. A diagram of the concept model for water management at Keness is shown on Figure 20-1.

Figure 20-1: Conceptual model for water management at Kemess



Treatment and Site-Wide Integration

All contact water from the North Main Zone Mining Area—including the pits and underground workings—will be captured and pumped to the south water management system for treatment. A new Water Treatment Plant (WTP) for the North Area or expansion of the exiting WTP will be studied in subsequent phases of the project. This maybe required to manage metals such as copper and cadmium, as predicted annual contact volumes will peak at approximately 2.5 Mm^3 by Year 13. In the South Area, the existing WTP will be restarted prior to first production, utilizing Selenium Ion Exchange (SeIX) treatment as assumed in the site-wide water model to meet permit limits.

Reclamation and Closure Water Management

The conceptual closure plan prioritizes transitioning to passive water management in perpetuity. The PAG WRSF will be regraded and capped with a low-permeability cover to limit oxygen ingress and meteoric water infiltration. Both the Main Zone Pit and KUG TSF will be allowed to flood passively to establish permanent water covers over reactive materials. Once water quality objectives are met, the KUG TSF will discharge via a new spillway to Kemess Creek, and the Main Zone pit lake will report to East Cirque Creek. Active treatment is conservatively assumed for the KUG TSF for approximately five years post-operations until discharge requirements are satisfied.

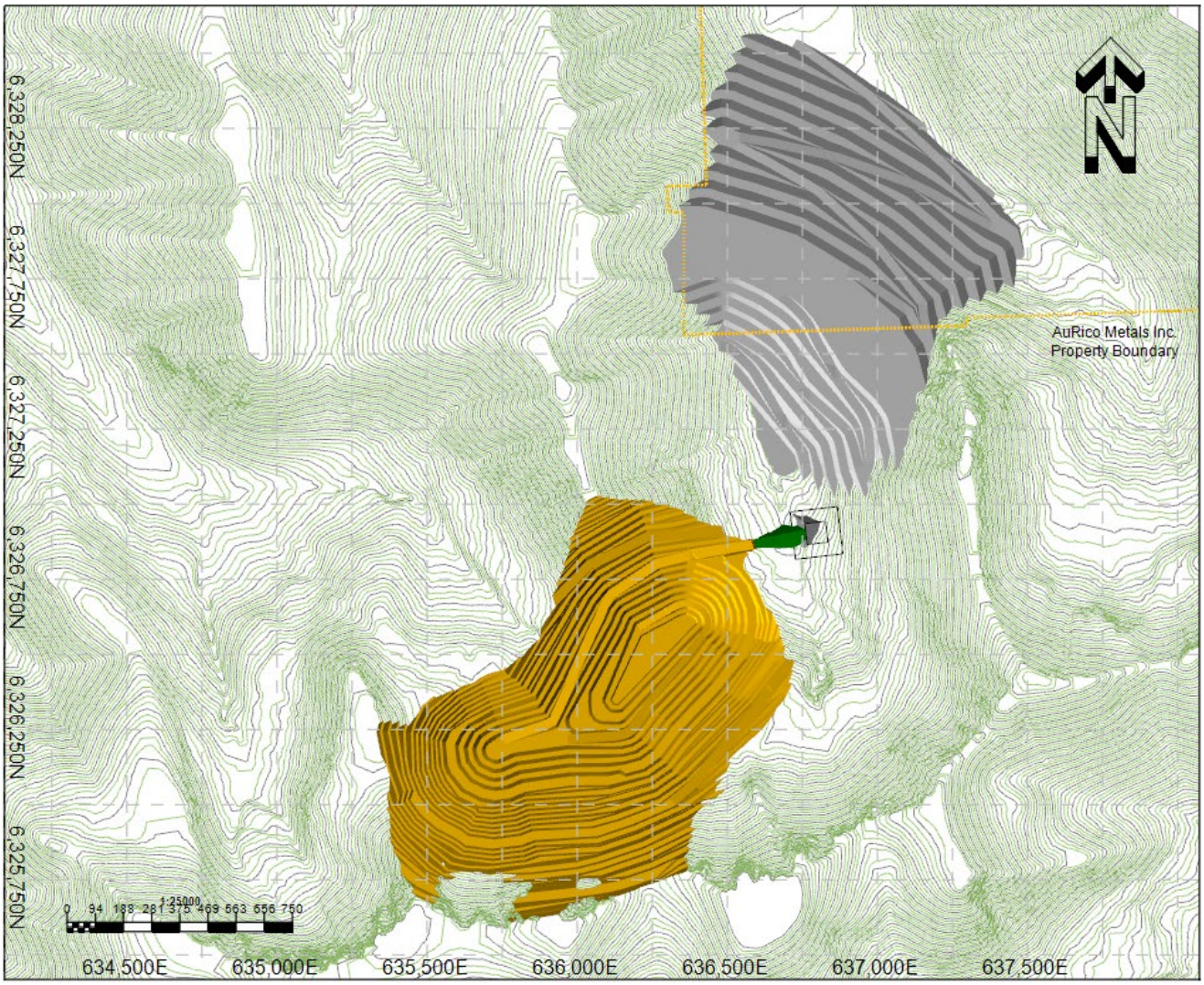
General Reclamation and Closure

At closure, both facilities will be transitioned to a state of long-term stability. The KUG TSF and KS TSF will feature permanent water covers to prevent the acidification of stored materials. The KUG TSF will eventually discharge passively via a new spillway to Kemess Creek, while the KS TSF will discharge to South Kemess Creek once water quality objectives are met.

Waste Rock Storage Facility

All Project infrastructure is located on land owned or administered by Centerra, except for a portion of the planned WRSF. The Company does not currently hold ownership of these lands and would be required to secure long-term surface access agreements or acquire the lands to support development. There is no assurance that such agreements can be obtained on acceptable terms and, if necessary, the WRSF will be relocated or redesigned to reside entirely on Centerra property which may result in additional costs.

Figure 20-2: Waste rock storage (gray) and Kemess property limit (yellow line)



20.3 PROJECT PERMITTING

The environmental assessment (EA) and permitting framework for mining in Canada, and British Columbia in particular, is well established, providing a comprehensive mechanism for reviewing major projects and to assess potential impacts. In addition, numerous policies and technical guidance documents exist and adherence to these is expected. The government-issued EA Decision Statement and Environmental Assessment Certificate as well as site operating authorizations and permits held by Kemess Mine followed rigorous and robust regulatory processes. These processes involved review by numerous ministries and agencies as well as indigenous nations and local communities

The permitting strategy centres on amending the project's existing robust authorization base to incorporate the transition from block cave mining to a combined open pit and underground operation. This approach allows the project to utilize "shovel ready" permits for immediate field activities—such as

refurbishing the mill, constructing the access tunnel, and dewatering the KUG TSF—while the more complex major amendments are processed.

Kemess Mine was previously subject to the Metal and Diamond Mining Effluent Regulations (MDMER) in connection with previous authorized discharges, however, the mine received a closed mine status letter from Environment Canada (EC) in 2014, excluding Kemess from the section of the MDMER regulation that require monitoring and reporting and instead making the mine subject to the Fisheries Act. However, future authorised discharges will result in a new registration of the Kemess mine under the MDMER regulation.

20.3.1 Concurrent Permitting and IAAC Integration

Centerra intends to employ a Concurrent Permitting model. This streamlined model, successfully utilized for the Mount Milligan 2035 Life-Of-Mine expansion, emphasizes deep interagency coordination to reduce regulatory uncertainty.

Crucially, this strategy requires the Impact Assessment Agency of Canada (IAAC) to be integrated into the concurrent model alongside provincial regulators, such as the Ministry of Mines and Critical Minerals (MCM) and the Ministry of Environment and Parks (ENV). Technical studies are being aligned with the Joint Application for Information Requirements (JAIR) to facilitate a coordinated review that satisfies both provincial Environmental Assessment Certificate (EAC) requirements and the need to amend the Federal Decision Statement. This synchronized approach ensures that federal oversight under the *Impact Assessment Act* is managed in parallel with provincial Mines Act and *Environmental Management Act* (EMA) amendments.

Centerra met with IAAC, Environment and Climate Change Canada (ECCC) and DFO in August 2025 to provide overviews of the Mt. Milligan 2045 future plans as well as the Kemess Re-Start Plans. Subsequently, Centerra provided the provincial regulatory agencies with contact information for the federal agencies at their request to start the coordination process. Both the IAAC and the province pointed to a Memorandum of Understanding (MoU) in place that could support a coordinated process. Centerra plans to continue this dialogue with provincial and federal agencies as the project nears closer to an Initial Project Description (IPD).

20.3.2 Staged Application Packages

The project has bifurcated its permitting requirements into two distinct packages to de-risk the execution timeline and manage engineering dependencies.

- Existing framework: The site currently holds Mines Act Permit M-206, EMA Effluent Permit PE-15335, and Air Emission Permit PA-109392. The existing regulatory framework for the Kemess site provides a robust foundation of “shovel-ready” authorizations that allow critical

field activities to proceed while KRP-specific amendments are processed. Under EMA Permit PE-15335 (Part 2) and Mines Act Permit M-206, the project is currently authorized to perform the dewatering of the KUG TSF (the former Kemess South Pit) at a target discharge rate of 10.3 Mm^3 per year to Attichika Creek. This framework also covers the 2026 restart of the seasonal Selenium Ion Exchange (SeIX) water treatment plant, which may be required for managing selenium concentrations during the drawdown period. Beyond water management, several major infrastructure and development activities supporting the KRP are already permitted and can be initiated immediately. These include the partial refurbishment of the processing mill (currently approved for 37,500 tpd), construction of the access tunnel between the Kemess South area and Kemess Lake Valley, and development of the triple decline portals and tunnels.

- Early Works Application (Q2 2027): This initial package will seek authorization for North Access Road upgrades, new bunkhouse, and the realignment of one of the three approved decline tunnels. Permit approvals are expected within one year of application submission.
- Major Works Application (Q2 2028): A combined Environmental Assessment Certificate (EAC)/Federal Decisions Statement and Mines Act/Environmental Management Act (EMA) amendment will cover the Main Zone and Nugget Pits, the PAG WRSF, the new leach plant, and the expanded KS TSF. Permit approvals are expected within one year of application submission.

20.3.3 Technical Path and Site Performance Objectives

Strategic technical programs have been front-loaded to prevent delays in the permitting process. A primary focus is currently on the development of a Site Performance Objective (SPO) for a long-term selenium permit limit, with a final submission deadline of March 2026. This SPO is supported by voluntary studies requested by Indigenous partners, including groundwater connectivity assessments and bird and amphibian egg monitoring, which are essential for addressing concerns regarding water quality in the Waste Rock Creek and Attichika Creek catchments.

20.3.4 Collaborative Indigenous Oversight

Under the existing Impact Benefit Agreement (IBA) with the Tse Keh Nay (Takla, Kwadacha, Tsay Keh Dene Nations), the Environmental Management Committee (EMC) serves as the primary mechanism for collaboration on all KUG Project-related environmental matters. Takla, Kwadacha, Tsay Keh Dene Nation members of the EMC participate in the collaborative development of permit applications and management plans, so that Traditional Knowledge and Takla, Kwadacha, Tsay Keh Dene Nation values have an opportunity to be integrated into the project design before submission to government agencies.

20.4 SOCIAL OR COMMUNITY REQUIREMENTS

Within the federal and provincial regulatory frameworks, consultation with Indigenous nations, local communities and other interested parties are required for new or amended mine project applications.

Centerra maintains a proactive and transparent relationship with its Indigenous partners Takla, Kwadacha and Tsay Keh Dene First Nations, and Gitxsan Nii Gyap First Nations. The closest communities to the Kemess mine site by air are Kwadacha (Fort Ware), Tsay Keh Dene, and Takla Landing.

- Kwadacha (Fort Ware): Located approximately 79 km from the project site, is the closest community to the mine.
- Tsay Keh Dene: Situated on the north end of the Williston Reservoir, this community is approximately 111 km away from the mine site.
- Takla Landing: The main community for the Takla Lake First Nation is located roughly 182 km from the project.

Impact Benefit Agreement (IBA): A comprehensive IBA was executed in May 2017, establishing a long-term framework for financial benefits, employment, and environmental oversight with Takla, Kwadacha and Tsay Keh Dene.

Collaborative management: Two key committees ensure Takla, Kwadacha and Tsay Keh Dene First Nations participation: the Environmental Management Committee (EMC), which reviews all permit applications, and the Business Opportunities Committee (BOC), which works to maximize Takla, Kwadacha and Tsay Keh Dene First Nations contracts. For evaluating bids, Centerra gives weight to Takla, Kwadacha and Tsay Keh Dene First Nations content.

Centerra also has an agreement with the Gitxsan Nii-Gyap First Nation that is specifically designed to provide benefits for permit review, cultural protection, and general capacity building. As a key stakeholder in the regional environmental framework, the Gitxsan Nii-Gyap are included in the consultative review process for essential site documents. To support their ongoing participation and community interests, the Kemess Restart Plan incorporates financial benefits to the Gitxsan Nii-Gyap throughout the duration of the project.

The relationship of the Project with local land users is formalized through the Trapline Holders Settlement and Release, an agreement originally executed on December 18, 2014, with the registered holders of Trapline No. 0739T006. This agreement is one of the foundational community and Indigenous contracts that support the Kemess Project and is focused on facilitating capacity building and participation in the regulatory review process. Under the terms of the Kemess Restart Plan, the Project

has committed to providing financial benefits to the trapline holders to mitigate impacts and support traditional land use.

Community concerns: During a leadership meeting in December 2025, Takla, Kwadacha and Tsay Keh Dene First Nations Chiefs emphasized concerns regarding selenium management, the transport and use of cyanide for the leaching circuit, and the effectiveness of water treatment systems. Centerra has committed to addressing these concerns within the framework of the existing IBA to address the restart plan's specifics.

Centerra is committed to continuing to build and maintain long-term, positive relationships with indigenous nations potentially affected by the Kemess Project. The company promotes ongoing communication, the sharing of information in an open, collaborative, and respectful manner, and values the incorporation of feedback from Indigenous nations and local communities. Also important is the extension of existing and emerging sustainable economic benefit opportunities that would align with the values and goals of Indigenous nations and local communities.

20.4.1 Engagement Planning

The engagement plan for the Kemess restart process centers on transparency, collaborative oversight, and proactive relationship management with the five Indigenous groups holding formal agreements: the Tse Keh Nay (Takla, Kwadacha and Tsay Keh Dene First Nations comprised of the Takla, Kwadacha, and Tsay Keh Dene Nations—the Nii-Gyap, and regional Trapline Holders.

Collaborative Governance Committees

The primary vehicle for engagement is the suite of joint committees established under the 2017 Impact Benefit Agreement (IBA), which will be utilized to manage the transition to the restart plan:

- Environmental Management Committee (EMC): Meets at least quarterly to collaboratively develop permit applications, review monitoring data, and integrate Traditional Knowledge into environmental management.
- Senior Implementation Committee (SIC): Comprised of First Nations Chiefs and Centerra executives to oversee the broad implementation of the IBA and resolve high-level disagreements.
- Business Opportunities Committee (BOC): Identifies upcoming contract opportunities and promotes the use of Takla, Kwadacha and Tsay Keh Dene First Nations businesses, applying weight to Takla, Kwadacha and Tsay Keh Dene First Nations content when evaluating project bids.

Direct Community Outreach and Communication

To ensure fact-based information flow to all community members, Centerra will implement the following:

- Annual community meetings: Hosting meetings in each Takla, Kwadacha and Tsay Keh Dene First Nations community to present project updates and answer questions.
- Plain language reporting: Distribution of newsletters, posters, and summaries that describe the restart plan and environmental safeguards without complex technical jargon.
- First Nations liaison: Hiring a dedicated site-based liaison to facilitate daily communication and collaboration.

Key Consultation Focus Areas (2026–2028)

Engagement throughout the restart will prioritize addressing specific community concerns validated during leadership meetings in late 2025:

- Water management: Extensive consultation on the Selenium Site Performance Objective (SPO) submission due in March 2026 and the restart of the Water Treatment Plant.
- Amazay Lake Protection: Demonstrating through technical modeling that the new mine plan has no impact on the Amazay Lake watershed.
- Waste and chemicals: Engaging on the management of potentially acid-generating (PAG) waste rock storage and the safe transport and use of cyanide in the new gold leach circuit.

Relationship Maintenance Strategy

To manage the risk of requests to reopen the IBA due to evolving regional expectations, the project will follow a proactive path:

- Capacity building: Providing financial benefits to the Gitxsan Nii-Gyap and the Trapline Holders to support their participation in the regulatory review process.

20.5 CLOSURE PROGRAM

Mine closure and reclamation planning for the Kemess Restart Project is being advanced in accordance with the regulatory requirements of the *Mines Act*, *Environmental Management Act*, and the Health, Safety and Reclamation Code for Mines in BC (HSRC). The conceptual closure plan is designed to restore the landscape for traditional land uses, such as hunting and fishing, while aiming to achieve passive water management in perpetuity. This plan builds upon the existing authorized framework, incorporating new mining components including the Main Zone and Nugget pits, the PAG WRSF, and the expanded KS TSF.

20.5.1 Waste Rock Storage Facility (WRSF) Closure

The PAG WRSF in the East Cirque Valley will be managed to promote long-term physical and chemical stability.

- Regrading: Facility slopes will be regraded to a planar 2H:1V configuration to match the surrounding topography.
- Cover system: A multi-layer low-permeability cover will be installed to limit oxygen ingress and meteoric water infiltration into the PAG material. This cover is assumed to consist of 1.0 m of earth-fill topped with 0.3 m of growth medium. Subsequent engineering studies will define specific low-permeability and earth-fill material quantities and sources.
- Water management: Diversion and collection channels will be upgraded to convey a 1-in-200-year, 24-hour storm event. Contact water captured from the facility during the flooding period will either be pumped back into the Main Zone Pit or treated until it is suitable for release into East Cirque Creek.

20.5.2 Pit and TSF Flooding

The long-term strategy for reactive materials involves establishment of permanent water covers to prevent acidification.

The flooding of the pits and TSFs occurs at different rates depending on the specific facility and the water management strategy applied during the closure phase.

Main Zone Pit Flooding

Numerical groundwater modeling indicates that once dewatering ends in Year 18, the Main Zone Pit will function as a net groundwater sink even after reaching its natural spill elevation of 1,571 masl, effectively preventing the migration of contact water into the surrounding environment.

The duration of this flooding depends heavily on whether contact water from other facilities is diverted into the pit:

- Passive flooding: Without any additional inputs (only pit wall runoff and groundwater), the pit is expected to take 45 years to reach the target elevation.
- With WRSF capture: If seepage and runoff from the WRSF are pumped into the pit, the filling period is reduced to 24 years.
- Accelerated flooding: If additional flows from the area between the open pit and the WRSF are included, the duration could be further reduced to approximately 11 years.

Kemess Underground (KUG) TSF Flooding

KUG TSF (existing KS Pit): This facility will maintain a permanent water cover over sub-aqueously stored cleaner tailings. It will eventually discharge passively via a new spillway (1,255 masl) into Kemess Creek rather than Waste Rock Creek.

- Active phase: The project assumes active water treatment will continue for 5 years post-operations to manage water levels and quality.
- Passive overflow: The facility is predicted to transition to passive overflow into Kemess Creek within 4 years after active treatment ceases, totaling approximately 9 years after the end of operations.

Kemess South (KS) TSF Flooding

The expanded KS TSF, which will have an updated storage capacity of 143.3 Mm^3 and a spillway elevation of 1,532 masl, is predicted to flood relatively quickly compared to the other facilities. This facility will maintain a permanent water cover over sub-aqueously stored cleaner tailings. At closure, beach slopes above the water line will be revegetated. Of consideration are:

- Overflow timing: The facility is predicted to begin passive overflow to South Kemess Creek within three years after the start of the closure period.
- Management considerations: Because this occurs so soon after closure, the project may increase mill makeup water withdrawals from the KS TSF during the final years of operation to reduce the free water inventory and allow more time for constituent concentrations to decline before passive discharge begins.

20.5.3 Post-Closure Water Treatment

While the ultimate goal is passive treatment, the KRP conservatively accounts for active treatment requirements during the transition phase.

- KUG TSF treatment: The project assumes active treatment for the KUG TSF will continue for five years post-operations to manage water levels and quality. The transition to passive discharge into Kemess Creek is expected to occur within four years after active treatment stops.
- Kemess North area: Contact water from the WRSF and pit highwalls will be treated until reclamation is complete and water quality meets permit criteria for release.
- Passive solutions: The project aims to research and implement passive treatment technologies in perpetuity for the KUG TSF and Main Pit, though active treatment remains a worst-case contingency for the cost model.

20.5.4 Financial Security and Assurance

Centerra has accounted for significant financial obligations to satisfy regulatory and community requirements.

- Closure costs: The estimated undiscounted cost of closure and reclamation is approximately CA$248 million (US$180 million), which includes CA$92 million for water quality mitigations.
- Financial assurance for major events: Under the 2017 IBA, the project must maintain a $10 million financial assurance (adjusted annually for inflation) to remediate any major accidents or malfunctions. This assurance is required from the start of tailings dam construction until the end of closure when the dam is certified as sound.
- Environmental indemnities: A separate $5 million security (adjusted for inflation) is maintained for general environmental indemnities.

21 CAPITAL AND OPERATING COSTS

21.1 OVERVIEW

All capital and operating cost estimates presented in this Item are preliminary in nature and prepared to an American Association of Cost Engineers (AACE) Class 5 level of accuracy (-50%/+100%) appropriate for a PEA study. These estimates are based on conceptual mine plans, benchmarked data, and assumptions considered reasonable for this stage of study. The LOM economic results are calculated from January 1, 2028, onwards, and therefore, all capital and operating costs are presented from the same timeline basis (January 1, 2028, onwards), however, all cost estimates are expressed in constant dollars as of January 1, 2026. It is assumed that the Project construction begins in 2028 with first production expected in late 2031. Total LOM costs are estimated at $6.9 billion, including $1.6 billion for capital costs and $5.2 billion in operating costs. A summary of total LOM capital and operating costs are detailed in Table 21-1.

Table 21-1: LOM capital and operating costs

Cost summary ($ M)	LOM total[1]
Operating costs	
Open pit mining	1,063
Underground mining	966
Processing	1,558
General and Administration	975
Transportation	422
On-site operating costs	**4,985**
Treatment and refining	131
Selling and marketing	98
Subtotal – operating costs	**5,213**
Capital costs	
Initial non-sustaining capital costs [NG]	771
Expansionary non-sustaining capital costs [NG]	277
Sustaining capital costs [NG,2]	595
Subtotal – capital costs	**1,643**
Total costs	**6,858**

Notes: (1) Totals may not sum due to rounding. NG: initial non-sustaining capital costs, expansionary non-sustaining capital costs, sustaining capital costs are non-GAAP financial measures. (2) Does not include capital lease payments of $229 million which are included in the economics and operating costs above.

Sensitivity analyses contemplating variations of input assumptions are provided in Item 22.4 of this document.

21.2 BASIS OF ESTIMATE

Cost estimates for the Kemess Mine were developed with reference to similar open-pit and underground mining operations. Any estimates made in Canadian dollars were converted to constant US dollars

applying an exchange rate of 1.38 (CA$:US$) as of January 1, 2026 as outlined in Item 22.1 (“Assumptions”). Estimates include direct and indirect costs, initial non-sustaining capital, expansionary non-sustaining capital, sustaining capital, and closure costs, but exclude financing charges and corporate overhead. All estimates of capital costs are presented in the US dollars and no allowance for escalation or exchange rate fluctuation were used.

21.3 CAPITAL COST ESTIMATES

Life-of-mine project capital costs are estimated to total $1.6 billion, comprising the following three distinct phases:

- Initial non-sustaining capital costs – $771 million
- Expansionary non-sustaining capital costs – $277 million
- Sustaining capital costs – $595 million.

Initial non-sustaining capital costs, expansionary non-sustaining capital costs, and sustaining capital costs incorporate 40% for indirect costs and 30% contingency applied to construction items.

Table 21-2: LOM capital costs[1] summary

Capital costs ($ M)	Initial non-sustaining	Expansionary non-sustaining	Sustaining capital costs	Total capital costs
Process plant	96	64	37	197
Open pit conveyor system	65	-	-	65
Open pit crusher	25	-	-	25
Open pit infrastructure and other	29	-	25	54
PAG storage facility	24	-	11	34
General site infrastructure	27	-	11	38
Tailings facility	9	-	89	98
Underground infrastructure	-	43	56	99
Underground development	-	59	38	97
Underground conveyor system	-	12	7	19
Paste backfill plant	-	-	75	75
Mobile fleet overhauls/replacements	-	3	101	103
Subtotal Infrastructure directs	**275**	**180**	**448**	**902**
Construction indirects	110	47	72	229
Contingency	115	50	76	241
Subtotal Infrastructure costs	**500**	**277**	**595**	**1,372**
Capitalized open pit pre-stripping	131	-	-	131
Capitalized pre-production G&A	101	-	-	101
Open pit conveyor tunnel	39	-	-	39
Total indirect and other costs	**271**	**-**	**-**	**271**
Total Capital Costs	**771**	**277**	**595**	**1,643**

Notes: 1. Initial non-sustaining capital costs, expansionary non-sustaining capital costs, sustaining capital costs are non-GAAP financial measures. 2. Totals may not sum precisely due to rounding.

Initial non-sustaining capital costs [NG] – this phase represents all expenditures required to develop the property and achieve the first open-pit production by late 2031. Infrastructure spending is planned over the three-year period from 2028 to 2030.

The initial capital cost of $96 million for the processing plant includes refurbishment of the Kemess mill grinding circuit ($57 million), comprising refurbishment of SAG and ball mills, installation of a new regrind circuit ($8 million), refurbishments of flotation circuits including rougher flotation ($19 million) and cleaner flotation ($6 million), and other mill equipment refurbishment works ($7 million).

The initial capital cost also includes construction of an 8 km conveyor system ($65 million), consisting of 3 km underground, and 5 km overland with 1 km of the overland portion running through a tunnel. The $39 million conveyor tunnel development cost covers excavation and construction of the 1 km tunnel through a mountain ridge.

Additional infrastructure costs include $25 million for a surface crusher and $29 million for the Kemess Main open pit infrastructure comprising truck shop, laydown area, power infrastructure and other supporting facilities. The $27 million general site infrastructure cost includes a personnel camp and road upgrades.

Construction of the PAG storage facility, located adjacent to the Kemess Main open pit is estimated to cost $24 million and includes the costs of a liner, construction of a pump station, and a water management pond.

Capitalized open pit pre-stripping costs total $124 million and includes initial waste pre-stripping in the Kemess Main open pit, along with a $7 million purchase of surface mobile fleet equipment which would not be subject to any lease arrangement. Capitalized pre-production G&A costs totalling $101 million includes general and administrative costs required during the development period, salary and wages of administrative employees, camp management, flight logistics, insurances, and other administrative costs. Both capitalized open pit pre-stripping costs and capitalized pre-production G&A costs apply to the 2028-2031 pre-production period.

Expansionary non-sustaining capital costs [NG] – this phase includes the capital required to continue developing the property following first production in late 2031 and to enable the ramp-up of the Kemess Underground (“KUG”) mine. Expansionary capital totals US$277 million and collectively supports completion of the leach plant, advancement of the underground mine, and expansion of key mine services necessary to reach the designed production profile. Expansionary capital is focused on enabling the transition from initial open-pit-supported production to a fully integrated 50,000 tpd open pit and underground operation from 2033 onwards.

Expansionary plant costs include the construction of the leaching circuit in 2032, which is expected to improve overall gold and silver recoveries and cost approximately $64 million excluding 40% indirect costs and 30% contingency ($116 million including indirect costs and contingency). Underground infrastructure construction costs total $43 million and will cover KUG infrastructure including workshops, electrical installations, ventilation infrastructure, dewatering systems, and related mine services. Underground development totals $59 million for ongoing lateral and vertical development to access deeper production stopes, establish additional levels, and support sustained UG mining rates of approximately 8 ktpd upon achieving steady-state production. A direct cost of $12 million has been estimated for construction of the UG crusher and UG conveyor,.

Sustaining capital costs [NG] – this phase includes all expenditures required to maintain safe and reliable operations of the open pit and underground mine, processing plant, tailings facilities, and supporting site infrastructure over the life of mine. Sustaining capital costs are estimated to total $595 million and cover asset replacements, major overhauls, and ongoing infrastructure required to keep mining and processing activities operating at planned rates.

Underground development costs of $38 million will cover ongoing lateral development, re-muck stations, and access required to sustain long-hole stoping production beyond the initial and expansionary development phases. Mobile fleet overhauls/replacements total $101 million allocated to underground ($53 million) and open pit fleets ($47 million). Construction of the paste backfill plant is estimated to cost $75 million. Open pit infrastructure includes $25 million for continuous open pit geotechnical drilling. Mill maintenance costs over the life of mine are estimated at $37 million for the replacement and refurbishment of processing equipment.

During the first three years of operations, tailings will be deposited into the previously mined Kemess South Open Pit. Water levels at the Kemess South Open Pit will be lowered prior to first production, allowing it to be used as the initial tailing storage facility (KUG TSF), which has sufficient capacity to support the first three years of operations. In the fourth year of operations, rougher tailings deposition will transition to the existing tailing storage facility (Kemess South TSF), which will be utilized for the remainder of the life of mine. The tailings from the leach plant will continue to be deposited to the KUG TSF. Sustaining capital of $89 million is required to complete the transition of rougher tailings deposition to the Kemess South TSF, and includes the construction of new water diversion dams, tailings delivery pipelines, and other associated civil works. Pumping costs for tailings delivery to both TSFs are included in processing cost estimates.

Not included in the capital costs are long-term agreements associated with lease contracts for equipment, rail cars, and concentrate storage. Lease payments are included as a separate line item in the production schedule and cash flow summary in Table 22-1 and are included in the project NPV figures. Capital lease payments of approximately $40 million, to be expended during the initial project

development period, are not included in the non-sustaining initial capital expenditures, but form part of the initial cash outflow prior to first production. Capital lease payments, including equipment and other leases, of approximately $229 million, made after first production, are treated as sustaining capital lease payments and are not included in the sustaining capital expenditure estimates. Sustaining capital lease payments are included in the determination of the LOM AISC for the project.

21.4 OPERATING COST ESTIMATES

Operating costs are estimated to total $5.2 billion over the LOM and include open pit mining, underground mining, processing, general services and administration (G&A), transportation, treatment and refining, and selling and marketing costs. A summary of operating and all-in sustaining costs ("AISC") is presented in Table 21-3:

Table 21-3: LOM operating costs

Cost summary	Average LOM ($/t processed)
Operating costs	
Open pit mining	4.17
Underground mining	3.79
Processing	6.11
General and administration (G&A)	3.83
Transportation	1.66
On-site operating costs	19.56
Treatment and refining	0.51
Selling and marketing	0.39
Total operating costs	**20.46**

The LOM operating cost per tonne processed is estimated to average $20.46. Gold production costs are estimated to average to $1,401 per gold ounce sold over the LOM. The all-in sustaining cost (AISC) estimate includes sustaining capital costs, sustaining lease payments, ARO accretion expense and by-product credits for net copper and silver sales, and is estimated to average $971 per gold ounce sold over the LOM. AISC is projected to be lower than the average gold production cost per ounce, as the total of copper and silver by-product revenue credits are anticipated to exceed sustaining capital expenditures.

21.4.1 Mining

The Kemess PEA outlines a development approach in which open pit mining begins first, followed by the start of underground production approximately two years later. Once underground production commences, both mining methods operate concurrently for the remainder of the projected 15-year mine life.

Mining operations will begin with waste pre-stripping at the Kemess Main open pit with production from the open pit planned to commence approximately two years later. Underground production will start approximately an additional two years after that. Production from underground is expected to achieve 0.8 Mt in Year 3 and 2.5 Mt in Year 4. Prior to the start and ramp up of underground operations, the open pit will supply process plant feed at a nominal average rate of 40,000 tpd, or 15 Mtpa from Years 2 to 3. Once both open pit and underground mines are operating concurrently, combined mining activities will provide process plan feed at an average rate of approximately 50,000 tpd, or 18 Mtpa from Year 3 onward.

The LOM mining schedule is presented in the Table 21-4.

Table 21-4: LOM mining production schedule

Mine Production	Units[1]	LOM Total	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046
Open Pit mine production																					
OP ore mined	Mt	**224**	-	-	-	3	14	17	16	15	15	15	16	16	16	16	16	16	16	16	2
OP waste mined	Mt	**168**	-	18	18	14	6	11	16	15	14	9	13	13	8	7	3	3	1	1	0
Open Pit material mined	**Mt**	**392**	**-**	**18**	**18**	**17**	**20**	**29**	**32**	**30**	**29**	**24**	**29**	**28**	**23**	**23**	**19**	**18**	**17**	**17**	**2**
UG mine production																					
UG ore mined	Mt	**31**	-	-	-	-	0	1	3	3	3	3	3	2	2	2	2	3	2	1	-
UG waste mined	Mt	**2**	-	-	-	1	1	0	0	0	0	0	-	-	-	-	-	-	-	-	-
UG material mined	**Mt**	**33**	**-**	**-**	**-**	**1**	**1**	**1**	**3**	**3**	**3**	**3**	**3**	**2**	**2**	**2**	**2**	**3**	**2**	**1**	**-**
Total mine production																					
Total ore mined	Mt	**255**	-	-	-	3	14	18	18	18	18	18	18	18	18	18	18	18	18	17	2
Total waste mined	Mt	**170**	-	18	18	14	7	12	16	15	14	9	13	13	8	7	3	3	1	1	0
Total material mined	**Mt**	**425**	**-**	**18**	**18**	**18**	**21**	**30**	**35**	**33**	**32**	**27**	**31**	**31**	**26**	**25**	**21**	**21**	**19**	**18**	**2**
Rehandle material moved	Mt	**2**	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total material moved	**Mt**	**427**	**-**	**18**	**18**	**18**	**23**	**30**	**35**	**33**	**32**	**27**	**31**	**31**	**26**	**25**	**21**	**21**	**19**	**18**	**2**

Notes:

1. “Mt” refers to millions of tonnes.
2. Totals may not sum due to rounding.

21.4.2 Open Pit Mining Costs

A summary of open pit (OP) mining costs is presented in Table 21-5:

Table 21-5: LOM open pit mining costs

Cost summary	Average LOM ($/OP tonne mined)
Open pit mining costs	
Direct OP mining	2.81
OP conveying and materials handling	0.22
Total OP mining costs	**3.03**

The LOM open pit mining unit cost estimate of $4.17/t processed equates to approximately $3.03 per open pit tonne mined which includes the unit cost of $2.81/t mined and $0.22/t mined (transportation via open-pit conveyor and rehandling). The mining rate of $2.81/t mined was developed primarily with the reference to comparable Canadian open pit mining operations, including the Company's Mount Milligan Mine, adjusted for incremental costs taking account the location of the Kemess mine and logistics.

21.4.3 Underground Mining Costs

A summary of underground (UG) mining costs is presented in Table 21-6:

Table 21-6: LOM underground mining costs

Cost summary	Average LOM ($/UG tonne mined)
Underground mining costs	
Direct UG mining	23.51
UG operational development	7.59
UG mineralized material transportation	0.38
Total UG mining costs	**31.48**

Underground mining will use longhole open stoping (LHS) with paste backfill which was determined to be the method most suited for the geometry of the Kemess Underground ("KUG") portion of the deposit. The life-of-mine underground total mining cost is estimated at $31.48 per underground mined tonne. This cost encompasses all underground production activities, including drilling and blasting, mucking and haulage, crushing and conveying, paste backfill deposition, labour, power, and mine services. The total UG mining cost includes direct mining costs as well as operational development and ore conveying costs. On a per tonne processed basis, total underground mining unit cost equates to approximately $3.79.

The UG cost estimate was developed through a combination of benchmarking against comparable Canadian underground operations and first-principles approach, and validated through an independent contractor review. This estimate incorporates cost estimates for:

- Operating activities, including drilling, blasting, mucking, haulage, crushing, conveying, ventilation, power, and labour.
- Operating development, which covers lateral development and drives, ramps, remuck bays, and associated ground support.
- Paste backfill deposition, which represents a major component of underground costs, scaled to maintain stope production and development, consistent with the PEA paste-plant design.

These costs form the basis of the underground mining cost structure and reflect the efficiencies of the LHOS method, the material-handling system, and the paste-backfill strategy.

21.4.4 Processing

After the processing plant refurbishments and upgrades, the Kemess plant is expected to achieve daily throughput production of 50,000 tpd, or 18 Mtpa. The updated flowsheet includes the construction of the leach plant, which is expected to increase overall gold recovery by approximately 14%, enhancing the project's economics. In addition to improving recovery, it is expected to provide flexibility by enabling the processing of mineralized material from potential satellite deposits in the future.

LOM plant processing schedule is presented in Table 21-7.

Table 21-7: LOM processing plant production schedule

Processing Production	Units[1]	LOM Total	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046
Ore processed	**Mt**	**255**	**1**	**16**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**17**	**2**
Gold feed grade	g/t	**0.39**	0.26	0.32	0.35	0.39	0.42	0.39	0.40	0.37	0.39	0.38	0.37	0.39	0.41	0.41	0.44	0.44
Silver feed grade	g/t	**1.26**	0.82	1.07	1.18	1.06	1.25	1.16	1.25	1.20	1.32	1.16	1.16	1.33	1.36	1.43	1.70	1.83
Copper feed grade	%	**0.18%**	0.10%	0.17%	0.18%	0.16%	0.19%	0.18%	0.18%	0.17%	0.18%	0.16%	0.17%	0.19%	0.19%	0.20%	0.22%	0.24%
Contained gold processed	Koz	**3,176**	11	165	204	228	248	226	234	215	229	220	217	230	239	243	242	24
Contained silver processed	Koz	**10,336**	36	544	685	623	736	682	733	704	777	681	679	778	800	840	941	98
Contained copper processed	Mlb	**1,017**	3	59	73	65	76	73	73	66	73	66	67	76	76	79	83	9
Gold recovery	%	**74.5%**	53.3%	53.3%	69.6%	73.8%	75.9%	75.9%	77.0%	76.7%	78.0%	77.5%	77.6%	77.0%	77.6%	76.6%	75.3%	74.3%
Silver recovery	%	**50.1%**	45.0%	45.0%	46.3%	51.2%	52.2%	52.5%	52.0%	51.2%	50.5%	51.6%	51.0%	50.5%	50.4%	49.7%	47.1%	45.0%
Copper recovery	%	**88.2%**	81.4%	81.4%	83.3%	86.2%	87.1%	87.3%	88.8%	89.5%	90.3%	90.0%	90.4%	90.1%	90.2%	90.0%	89.5%	89.3%
Gold recovered	Koz	**2,376**	6	88	142	168	189	172	180	165	178	171	168	177	185	186	182	18
Silver recovered	Koz	**5,171**	16	245	317	319	384	358	381	360	393	351	347	393	403	417	443	44
Copper recovered	Mlb	**898**	3	48	61	56	66	64	65	59	66	59	61	68	69	71	75	8
Dry concentrate produced	Kdmt	**1,939**	6	105	131	121	143	138	140	128	142	128	131	148	149	153	161	17
Gold payable produced	**Koz**	**2,323**	**6**	**86**	**139**	**166**	**185**	**168**	**176**	**162**	**175**	**167**	**165**	**174**	**181**	**182**	**179**	**14**
Silver payable produced	**Koz**	**4,654**	**15**	**220**	**285**	**287**	**346**	**322**	**343**	**324**	**353**	**316**	**312**	**353**	**363**	**376**	**399**	**40**
Copper payable produced	**Mlb**	**851**	**2**	**46**	**57**	**53**	**63**	**60**	**61**	**56**	**62**	**56**	**57**	**65**	**65**	**67**	**71**	**7**

Notes:

1. “Mt” refers to millions of tonnes; “koz” to thousands of ounces; “Mlb” to millions of pounds; and “Kdmt” to thousands of dry metric tonnes.
2. Totals may not sum due to rounding.

21.4.5 Processing Costs

Processing cost estimates presented in this section are based on the existing Kemess process plant and the refurbishments required to return it to operational status, supported by benchmarked data, and assumptions considered reasonable for this stage of assessment. The confidence level of the estimates of processing costs is enhanced by the fact that the Project already has the major processing facilities, structural components, and plant layout in place, which are aimed to reduce execution risks relative to a greenfield construction project. Processing costs were developed from basic cost elements including labour, reagents, consumables, power rates, water treatment costs, and equipment productivities. The process plant is designed for a nameplate feed rate of 50,000 tpd, which is assumed to be achieved from 2033 onwards.

The LOM processing cost is estimated to average approximately $6.11/t processed. The processing cost includes tailings pumping costs and leach plant operating costs. Electrical power costs were estimated based on expected consumption and power rates calculated by a third-party consultant from first principles using an estimated project load list. Consumption estimates and pricing for grinding media and reagents are based on metallurgical test work results, and internal procurement data collected from the Mount Milligan Mine, supplemented with historical data from similar operations in British Columbia, and adjusted for transportation costs to the Project site.

21.4.6 General and Administration Costs

General and administration (G&A) costs include all site-based support services required to operate the Kemess Mine and excludes any allocation of corporate G&A. These costs cover camp operations, flight logistics, site security, water treatment, environmental services, insurance, property taxes, permitting expenses, IBA commitments and other general and administrative functions not captured within mining, processing, treatment, or transportation cost categories.

The average G&A cost is estimated at approximately $3.83/t milled over the life of mine (LOM). The G&A costs include site general costs of $2.82/t milled and the IBA commitments cost that contributes approximately $1.01/t milled. Water treatment costs included in the G&A costs are estimated to cost $3.6 million per year starting from 2037 to 2045, contributing approximately $0.13/t milled over LOM. Labour costs were developed based on projected workforce requirements and expected labour rates for northern British Columbia. Insurance, permitting, and property taxes were benchmarked to the current costs at the Company’s Mount Milligan Mine, reflecting comparable jurisdictional and operational conditions. Third-party service costs including camp management, catering, charter flight operations, site security, water treatment, environmental monitoring, and other professional services were estimated using vendor quotes obtained for major cost categories and based on the projected scope of support services required during operations.

21.4.7 Closure and Post Closure Costs

Closure and post-closure activities will begin progressively following the end of production in 2046, with the majority of physical reclamation and earthworks planned for the two years immediately following cessation of production operations.

On an undiscounted and uninflated basis, LOM closure and reclamation cash costs are estimated to be approximately $180 million. For purposes of the economic analysis, reclamation expenditures were discounted back to January 1, 2046, representing both the end of mine operations and the assumed start date for mine closure, and incorporated into the cash-flow model, with the discounted cost to January 1, 2046 representing approximately $100 million. This amount is then further discounted to the Project valuation date of January 1, 2028 within the cash-flow model, consistent with the treatment of all other cash flows.

While ongoing post-closure activities for monitoring, water management, and maintenance of closure works extend beyond 2048, the present value of these expenditures is fully captured within the discounted closure estimate used in the economic analysis.

22 ECONOMIC ANALYSIS

Economic estimates presented herein are preliminary in nature and have been prepared to an AACE Class 5 level of accuracy (-50% / +100%), appropriate for a PEA study.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

22.1 ASSUMPTIONS

The economic analysis of the project was conducted using the following assumptions and basis:

- Project economics are based on a valuation date of January 1, 2028. The economic assessment employs a discounted cash flow (DCF) approach, with cash flows assumed to occur at the mid-year of each period. The net present value (NPV) is calculated by discounting the LOM cash flows from January 1, 2028 through the end of the LOM at a discount rate of 5%. Economics include the time value of money benefit of pushing out $69 million of care and maintenance and closure costs to the end of the LOM.
- Open pit mining activities scheduled from January 1, 2029, followed by the start of underground development, in 2031. Processing will begin in late 2031 and continue to the end of mine life in 2045. The mine life counted from the start of processing in late 2031 to 2045 is estimated to be 15 years.
- All costs presented are in constant United States dollars as of January 1, 2026, with no price inflation or escalation factors applied.
- The metal price assumptions for gold, copper, and silver, and the US dollar to one Canadian dollar exchange rate used in the evaluation of the project economics are as follows:
  - Gold price per troy ounce: $3,000
  - Copper prices per pound: $4.50
  - Silver prices per troy ounce: $37.50
  - US dollar to one Canadian dollar exchange rate: $1.38.
- The silver produced is subject to a stream arrangement with Triple Flag. The impact of the stream arrangement is fully incorporated into the project economics.

- Working capital for the mine is assumed not to change significantly over the LOM and is not modelled in this economic analysis.
- No salvage values are assumed for the capital equipment at the end of mine life.
- Transportation costs include estimated delivery costs of gold and silver doré, and copper concentrate to customers.

22.2 TAXATION

The determination of taxes involves significant estimation and judgment requiring a number of assumptions. The actual taxes payable for the project will be subject to assessments by taxation authorities who may interpret tax legislation differently. The after-tax cash flow is based on best estimates by Company management of the probable outcome of these matters.

22.2.1 Corporation Income Taxes

Based on the pricing assumptions noted above, the Kemess Mine is expected to be subject to Canadian federal and British Columbia provincial corporate income taxes at the combined statutory rate of 27%. Corporate income tax liabilities can be partially reduced by available tax deductions, which are expected to partially offset taxable income. These deductions include:

- Exploration expenditures allowed to be claimed discretionarily at 100%, limited to the mine's taxable income
- Pre-production development expenditures allowed to be claimed discretionarily at 30% of the year-end balance
- Initial and sustaining capital expenditures generally allowed to be claimed discretionarily at 25% of the year-end balance
- Net operating loss carry-forward allowed for up to 20 years
- Provincial mining taxes.

Following the utilization of these deductions, the Kemess Mine is expected to be subject to corporate income taxes at the statutory rate.

22.2.2 British Columbia Mining Taxes – Provincial

The mine will be subject to the greater of two different taxes: either 2% tax on net current proceeds (net revenue less operating costs) or 13% tax on net revenue (net revenue less operating costs and capital expenditures). Based on the pricing assumptions noted above, the mine is expected to pay the 2% net current proceeds tax for approximately the first half of its life, as the Company expects to have sufficient deductions and credits during that period to offset the 13% tax on net revenue, while the mine is

expected to pay the 13% tax on net revenue for the second half of its life. In lieu of allowing a deduction of debt financing costs, the net revenue can be reduced by an investment allowance which is earned on expenditures incurred to the extent they have not yet been deducted.

22.3 LIFE-OF-MINE CASH FLOW FORECAST

The net undiscounted cash flows for the Kemess Mine from January 1, 2028 to the end of 2048 are estimated at $2,329 million, as presented in Table 22-1. The after-tax NPV of the LOM cash flow, discounted at 5% is estimated at $1,094 million.

Table 22-1: Cash flow summary (in millions of US dollars, unless otherwise indicated)

Description	Units	LOM Total	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047
Assumptions																						
Gold price	$/oz		3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000
Silver price	$/lb		37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50
Copper price	$/lb		4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50
USD to CAD Exchange rate	USD/CAD		1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38
Metal Sales																						
Gold ounces sold	Koz	**2,323**	-	-	-	6	86	139	166	185	168	176	162	175	167	165	174	181	182	179	14	-
Copper pounds sold	Mlb	**851**	-	-	-	2	46	57	53	63	60	61	56	62	56	57	65	65	67	71	7	-
Revenues																						
Gold revenue	$	**6,970**	-	-	-	18	257	418	497	554	504	528	485	524	501	494	521	544	547	536	42	-
Silver revenue[1]	$	**13**	-	-	-	0	-	-	1	1	1	1	1	1	1	1	1	1	1	1	0	-
Copper revenue	$	**3,829**	-	-	-	11	206	258	239	283	272	276	253	280	253	258	292	293	303	318	33	-
Smelting and refining costs	$	**(131)**	-	-	-	(0)	(5)	(6)	(7)	(8)	(8)	(9)	(8)	(10)	(9)	(10)	(11)	(12)	(13)	(14)	(1)	-
Total revenue	**$**	**10,681**	**-**	**-**	**-**	**29**	**459**	**670**	**729**	**830**	**768**	**797**	**731**	**795**	**747**	**743**	**802**	**826**	**839**	**841**	**74**	**-**
Silver stream upfront payments[1]	$	**45**	10	10	13	13	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total inflows	**$**	**10,726**	**10**	**10**	**13**	**41**	**459**	**670**	**729**	**830**	**768**	**797**	**731**	**795**	**747**	**743**	**802**	**826**	**839**	**841**	**74**	**-**
Unit Costs[2]																						
Gold production costs	$/oz	**1,401**	-	-	-	2,995	1,270	1,504	1,620	1,432	1,515	1,403	1,553	1,424	1,438	1,438	1,320	1,296	1,250	1,129	1,061	-
AISC[NG]	$/oz	**971**	-	-	-	4,285	1,606	1,589	1,741	1,428	1,189	900	1,066	836	895	877	610	601	492	205	43	-
Outflows																						
Operating costs	$	**(4,985)**	(0)	(0)	(0)	(29)	(195)	(336)	(394)	(396)	(388)	(373)	(378)	(377)	(358)	(357)	(353)	(357)	(350)	(317)	(26)	-
Selling and marketing costs[3]	$	**(98)**	-	-	-	(0)	(5)	(7)	(6)	(7)	(7)	(7)	(6)	(7)	(6)	(7)	(7)	(7)	(8)	(8)	(1)	-
Initial non-sustaining capital costs[NG]	$	**(771)**	(151)	(271)	(292)	(58)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Expansionary non-sustaining capital costs[NG]	$	**(277)**	-	-	-	(91)	(186)	(1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital costs[NG]	$	**(595)**	-	-	-	-	(109)	(93)	(78)	(89)	(17)	(41)	(27)	(26)	(24)	(24)	(19)	(20)	(17)	(10)	(1)	-
Lease payments	$	**(269)**	-	-	(25)	(21)	(28)	(35)	(41)	(44)	(48)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	-	-
Reclamation expenditures	$	**(100)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(36)	(36)
Cash taxes	$	**(1,302)**	-	-	-	-	(5)	(7)	(7)	(68)	(69)	(86)	(72)	(100)	(118)	(121)	(146)	(155)	(164)	(179)	(6)	-
Total outflows	**$**	**(8,397)**	**(151)**	**(271)**	**(317)**	**(199)**	**(529)**	**(478)**	**(525)**	**(605)**	**(529)**	**(510)**	**(487)**	**(514)**	**(509)**	**(510)**	**(529)**	**(542)**	**(541)**	**(517)**	**(70)**	**(36)**
Net cash flow	**$**	**2,329**	**(141)**	**(261)**	**(304)**	**(157)**	**(70)**	**191**	**204**	**225**	**239**	**287**	**244**	**281**	**238**	**233**	**273**	**284**	**298**	**325**	**4**	**(36)**
NPV @ 5%	**$**	**1,094**																				

Notes:

1. Silver revenue and silver stream upfront payments reflect the impact of the Kemess Silver Stream.
2. Unit costs average over LOM.
3. Selling and marketing costs include ocean freight.

"Mt" refers to millions of tonnes; "koz" to thousands of ounces; "Mlb" to millions of pounds; "kdmt" to thousands of dry metric tonnes; and "NG" to non-GAAP financial measure. NOTE: Totals may not sum precisely due to rounding.

22.4 SENSITIVITY ANALYSIS

Based on the PEA long-term gold and copper price assumptions of $3,000/oz and $4.50/lb, respectively, the Project's economics are estimated to generate $NPV_{5\%}$ of $1.1 billion and IRR of 16%.

Table 22-3 summarizes the sensitivities of Project NPV discounted at 5% to ±5% and ±10% changes from the base case gold and copper prices, foreign exchange, capital expenditures and operating cost assumptions.

Table 22-2: After-tax $NPV_{5\%}$ sensitivities to changes in assumptions

Millions of US dollars[1]	-10%	-5%	PEA	5%	10%
Gold price	848	971	1,094	1,216	1,338
Copper price	958	1,026	1,094	1,161	1,229
Canadian Dollar	782	946	1,094	1,226	1,346
Capital costs	941	1,017	1,094	1,170	1,246
Operating costs	903	999	1,094	1,188	1,283

Note: 1. Sensitivities are assumed constant for the LOM, flexed for each scenario, with the other assumptions the same as the PEA economics.

The analysis indicates that NPV is most sensitive to variations in the US dollar to one Canadian dollar exchange rate, gold price and gold recovery rate, followed by copper price and copper recovery, and initial capital expenditures. Variations in mining operating costs have the least impact of the contemplated parameters.

23 ADJACENT PROPERTIES

Several mineral properties and claims are located adjacent to or in the vicinity of the Project. Publicly available information indicates that these properties are at various stages of exploration and development.

Information regarding adjacent or nearby properties has not been used in the preparation of this Technical Report.

Mineralization, resources or reserves reported on adjacent or nearby properties are **not necessarily indicative** of mineralization on the Project.

24 OTHER RELEVANT DATA AND INFORMATION

The QPs are not aware of other data or information relevant to the Project.

25 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGICAL UNDERSTANDING

The Mineral Resource estimate for the Kemess Project is subject to uncertainties typical of a PEA-level study. Geological risks include uncertainty in the continuity of mineralization between drill holes, particularly in areas of wider drill spacing, along interpreted structural controls, and at depth, which may affect the confidence in resource classification and the geometry of the mineralized domains.

25.1.1 Lithology Model Risk

The QP considers the lithological model to be robust for both Kemess Main Zone and Kemess South models, as the principal lithological units are characterized by clear and readily distinguishable geological and physical contrasts in core and logging data. These contrasts support consistent lithological interpretation and domain definition at the current level of study. However, lithological interpretations may be refined as additional drilling is completed.

25.1.2 Fault Model Risk

The fault models for the Kemess Main Zone and Kemess South includes a network of faults identified within the deposit area. Certain interpreted faults have not been incorporated into the resource domain model, as they are not currently considered to be laterally continuous over significant distances or to materially affect mineral continuity based on available data. The interpretation of some faults is inherently subjective and dependent on drill hole orientation, drill spacing, and whether individual drill holes intersect fault structures. In addition, faults are currently modelled as 2D planes rather than three-dimensional volumes, and fault thicknesses are therefore not well constrained.

Minor faults excluded from the current model are not currently expected to have a material impact on grade distribution; however, the presence of additional, thicker, or more continuous faulting could affect mineral continuity, domain geometry, and the classification and quantity of the Mineral Resource estimate. As a result, future refinements to the structural interpretation may lead to revisions to the model.

25.1.3 Alteration Model Risk

The Kemess Main Zone alteration model is based primarily on geological observations, where potassic alteration is supported by magnetic susceptibility data and spectral analysis. Alteration boundaries are inherently gradational and do not represent fixed contacts. While the overall extent of the potassic alteration shell is considered reasonably well defined at the current level of study, differentiation between quartz–sericite–pyrite (QSP) and chlorite–sericite (C–S) alteration can be subjective where alteration intensity varies. CSA Global has completed a geochemical alteration model that is generally consistent

with the in-house model, although differences in alteration groupings and level of detail exist. As a result, further refinement of the alteration model with additional drilling and integrated interpretation may result in revisions to alteration domains and the Mineral Resource estimate.

The current geological model for Kemess South does not incorporate a detailed alteration model and is limited to the interpretation of a supergene enrichment zone. The absence of an alteration model introduces uncertainty in the geological interpretation and may result in future revisions to geological domains and the Mineral Resource estimate as additional drilling and alteration characterization are completed.

25.1.4 Resource Estimation Risks

The Kemess Main Zone Open Pit and Underground resource estimate carries inherent risks typical of early-stage projects, particularly due to the absence of production data. Key assumptions and confidence levels are primarily supported by geological interpretation and variogram-based continuity in the absence of production data to support reconciliation. Uncertainty remains until mining begins, as actual grade distribution, dilution, and recovery may differ from model predictions.

The Kemess South resource estimate benefits from historical operational and production data. Areas of the model that have been depleted show reasonable reconciliation with the current geological interpretation, lending further confidence to the estimate in previously mined domains.

Sampling-related risks, including potential bias, representativity, and density assumptions, are also recognized. The mineralization has been modelled as a continuous, single-phase system and is not expected to present significant additional geological complexity; however, it is recommended that the estimate be further validated through ongoing drilling, model updates, and eventual production performance data to enhance confidence in future reporting.

25.2 MINING

The concept of an open pit and an underground mine to optimally extract the maximum mineralized material from the Kemess Main Zone deposit is an economically viable option. Advanced engineering studies will seek optimized techniques and sequencing.

The main risks to the mine plan are geotechnical, given the unknowns associated with this level of study. Underground infrastructure locations have evolved from the previous block cave feasibility study. While historic information is available, the review and re-interpretation of this information is ongoing. Gaps in data and interpretation will need to be closed at the next phase of study.

The open pit mine is located in a topographically challenging area with risks that include a talus slope, natural rock falls and snow avalanches. While this PEA has concluded that removing the loose material

of the talus slope is the best way to manage that risk, life-of-mine geohazard management will be required.

Infrastructure congestion is another risk that will require additional study at the PFS level. Surface infrastructure locations will need to be located optimally between avalanche, terrain, and blast radius hazards.

Underground productivity is viewed as an opportunity for the Kemess Project. The current production limitation is the capacity of the paste backfill plant. The stope sizing, deposit geometry, and efficiency of the underground design are all viewed as comparable to other globally benchmarked longhole operations in the 10–12 ktpd range. The underground mining rate will be revisited as further study of the KUG longhole mine progresses.

25.3 METALLURGY AND MINERAL PROCESSING

Metallurgical testwork for the Main Zone and Kemess Underground deposits shows that a conventional sulphide flotation flowsheet with a primary grind of ~150 µm and regrind of ~20 µm is appropriate for the project. Main Zone composites achieved copper recoveries of 83–90% and gold recoveries of 54–59%, while Kemess Underground material demonstrated higher recoveries, particularly for copper. Mineralogical analysis shows chalcopyrite as the dominant copper mineral and pyrite as the primary gangue sulphide, with gold frequently locked within pyrite—supporting the inclusion of a leach circuit to recover gold from cleaner-scavenger tails.

Comminution testing reveals meaningful variability between Broken and Not-Broken Main Zone mineralized material types, with A×b values ranging from 40–69 and Bond BWi values of 16–18 kWh/t, indicating material hardness differences that will influence plant throughput. Additional variability testing is recommended to refine grinding energy estimates and support detailed design.

Deleterious elements in both feed and concentrate are low and well below penalty thresholds, with silica in Broken Zone concentrate being the only parameter requiring attention through blending. Cyanide leach tests on cleaner tails produced 55–76% gold extraction, validating the assumed 70% leach recovery used for PEA modelling.

Collectively, the testwork supports the selected flowsheet, highlights the importance of mineralization-type-based modelling, and identifies targeted areas for further metallurgical and comminution testing in the next project phase.

25.4 INFRASTRUCTURE CONCLUSIONS

Road upgrades are required north of the Kemess plant site to accommodate the movement of mine haul trucks to and from the Main Zone pit operation. Bridge upgrades are necessary but only for use by tare trucks.

The loadout facility at Mackenzie will need to be upgraded to be able to simultaneously service the Kemess mine and the Mount Milligan mine.

The existing personnel camp will be refurbished and additional modules installed to accommodate the anticipated larger workforce.

A new infrastructure pad is required at the Kemess North site to accommodate a truck shop, fuel station, paste backfill plant, emergency station, and offices. Other installations at the North site include a communications network and security facilities.

The paste backfill system requires additional thickener tanks at the mill and a 7 km pipeline to the KUG portal where the paste fill plant and binder storage tank will be constructed. An underground booster system will be constructed in the decline to pump the paste fill through the 3.6 km decline to the underground mine distribution system.

The main substation supplying the transmission line will require upgrades to support power delivery to both the Kemess Project and the Mount Milligan mine. Engagement with BC Hydro is ongoing to confirm upgrade requirements and to secure sufficient electrical power capacity for the Project.

A new 13.8 kV distribution line will be constructed to supply power to the Kemess North and underground mining areas. The final routing of this line will be determined in the next phase of studies and is expected to generally follow the conveyor alignment.

The Waste Rock Storage Facility (WRSF) is designed with relatively conservative slope stability assumptions. However, it is not certain that permission to construct the current configuration on property owned or administered by third parties will be obtained under acceptable terms. As there are alternative locations and configurations, this is deemed a low overall risk to the project.

The KUG TSF and KS TSF facilities are capable of storing the proposed quantity of tailings. The final elevation of the KS TSF will be approximately 1,537 masl with a 50 m extension and raise to 1,417 masl for the current buttress. The KUG facility will be filled to maximum tailings elevation of 1,257 masl.

Stability analyses were completed for the KS TSF embankments at their respective ultimate design elevations. Analyses used geotechnical material parameters from previous stability assessments which were based on historic laboratory testing, site investigations, and construction material testing. All dams

were found to meet or exceed the minimum factors of safety outlined by the Canadian Dam Association (CDA) and the Health Safety and Reclamation Code (HSRC) for Mines in BC.

Capital allocations for identified infrastructure installations and upgrades have been included in the Project capital cost estimate.

25.5 ENVIRONMENTAL AND SOCIAL

The Kemess Project possesses both provincial and federal environmental assessment (EA) authorizations, an EA certificate from the British Columbia environmental assessment office (EAO) and a Federal Decision Statement (DS) from the Impact Assessment Agency of Canada. The mine also possesses an Fisheries and Oceans Canada Authorization (FCA) from the federal Fisheries and Oceans Canada (DFO), and provincial Mines Act (MA) and Environmental Management Act (EMA) permits from the British Columba Ministry of Energy, Mines and Low Carbon Innovation. Finally, Kemess Mine also possesses ancillary permits from the British Columbia Ministry of Water, Lands and Resource Stewardship.

A streamlined permitting process will be pursued to complete necessary amendments to existing permits and authorizations to accommodate the project revisions described in this PEA

New environmental studies are required to support the environmental assessment for permitting the KRP. These studies include, but are not limited to, water quality and quantity, fisheries and aquatic resources, archaeology, soils, vegetation, and wildlife. Field work to complete the environmental studies commenced in 2024. with the aim to have baseline collection completed by the end of 2027/early 2028.

Modeling of surface water and water quality has confirmed that all contact water from the Main Zone Area—including the Main Zone Pit, Nugget Pit, and the PAG WRSF—will be captured and routed south to the existing treatment infrastructure, ensuring no impacts on Amazay Lake. The water balance model predicts that annual contact water volumes will rise from 1.1 Mm^3 in Year 1 to a peak of approximately 2.5 Mm^3 by Year 13.

To address predicted metal concentrations from mining activities a new Water Treatment Plant is planned in the East Cirque Creek catchment. Also, a Site Performance Objective is currently being developed to establish long-term selenium permit limits for Waste Rock Creek.

The Kemess site possesses an extensive biological database for fisheries and aquatic resources, with continuous data collection dating back to 1992 for sediment quality and fisheries resources. This long-term monitoring satisfies commitments under the Fisheries Compensation Agreement and the Fish and Aquatic Effects Monitoring Plan. The permitting strategy for Kemess centres on amending the project plan of operation from an underground block cave mine to an open pit and underground blasthole operation. This approach allows the project to utilize “shovel ready” permits for immediate field

activities—such as refurbishing the mill, constructing the access tunnel, and dewatering the KUG TSF—while the more complex major amendments are processed.

Centerra will employ a Concurrent Permitting model, successfully utilized for the Mount Milligan 2035 life-of-mine expansion, which emphasizes deep interagency coordination to reduce regulatory uncertainty. The strategy requires the Impact Assessment Agency of Canada (IAAC) to be integrated into the concurrent model alongside provincial regulators, such as the Ministry of Mines and Critical Minerals and the Ministry of Environment and Parks (ENV). This synchronized approach ensures that federal oversight under the *Impact Assessment Act* is managed in parallel with provincial MA and EMA amendments.

The site currently holds Mines Act Permit M-206, EMA Effluent Permit PE-15335, and Air Emission Permit PA-109392. The existing regulatory framework for the Kemess site provides a robust foundation of "shovel-ready" authorizations that allow critical field activities to proceed while KRP-specific amendments are processed. The Project is currently authorized to perform the dewatering of the KUG TSF (the former Kemess South Pit) at a defined discharge rate.

Both the Kemess South TSF and KUG TSF are permitted for the deposition of tailings.

The proposed PAG Stockpile will be a lined, sub-aerial dry-stack facility located in the East Cirque Valley, northeast of the proposed Kemess Main Zone pit designed to provide secure and permanent storage for approximately 158.5 Mt (72 Mm^3) of potentially acid-generating waste rock over a 17-year operating life. The WRSF design features a sophisticated dual-layer water management system designed to capture seepage and meteoric water - a collection system above the liner to collect meteoric water percolating through the waste rock and a drain system beneath the liner as a contingency system. Both systems direct seepage through a network of perforated pipes toward the North Collection Pond from which water will be pumped to a treatment facility.

An Early Works Permit application will seek authorization for North Access Road upgrades, bunkhouse renovations, and the realignment of one of the three approved decline tunnels. A Major Works Permit application will seek a combined Environmental Assessment Certificate (EAC)/Federal Decisions Statement and Mines Act/Environmental Management Act (EMA) amendment to cover the Main Zone and Nugget Pits, the PAG WRSF, the new leach plant, and the expanded KS TSF.

Each of the Works permit applications are expected to take 12 months for approval.

At mine closure, while the ultimate goal is passive treatment, the KRP plans, and has allocated costs, for active treatment during the transition phase, including KUG TSF water treatment for five years post-operations and the treatment of contact water from the WRSF and pit highwalls at the Main Zone mining area until reclamation is complete and water quality meets permit criteria for release.

Under the existing IBA with the Tse Keh Nay (Takla, Kwadacha an Tsay Keh Dene Nations), the EMC serves as the primary mechanism for collaboration on all KUG Project-related environmental matters. Takla, Kwadacha and Tsay Keh Dene Nations members of the EMC participate in the collaborative development of permit applications and management plans, ensuring that Traditional Knowledge and the values of the Takla, Kwadacha an Tsay Keh Dene Nations are integrated into the project design before submission to government agencies.

Centerra maintains a proactive and transparent relationship with its Indigenous partners Takla, Kwadacha and Tsay Keh Dene First Nations, and Gitxsan Nii Gyap First Nations. A comprehensive IBA was executed in May 2017, establishing a long-term framework for financial benefits, employment, and environmental oversight with Takla, Kwadacha and Tsay Keh Dene.

Centerra also has an agreement with the Gitxsan Nii-Gyap First Nation that is specifically designed to provide benefits for permit review, cultural protection, and general capacity building. The relationship of the Project with local land users is formalized through the Trapline Holders Settlement and Release, an agreement originally executed on December 18, 2014, with the registered holders of Trapline No. 0739T006.

A number of committees have been struck to coordinate collaborative governance, community outreach and communications with regional residents.

26 RECOMMENDATIONS

26.1 GEOLOGY AND EXPLORATION

The Kemess deposits are categorized as tilted tabular calc-alkalic copper-gold porphyry deposits. Recent exploration by Centerra has continued to define the Kemess North mineralization trend, an east-west belt of porphyry deposits, over 3.8 km in length, with variable depths from surface. From west to east the trend includes the Nugget zone, KUG deposit, Kemess Offset zone, Kemess East deposit, and Hilda South targets. There is a general trend of deposits occurring at greater depths towards the east.

High priority brownfield exploration and infill targets for resource growth are along the western and eastern margins of the updated resource shells, including the Nugget and Kemess Offset zones. The Kemess Offset zone between Kemess Underground and Kemess East should be drilled to identify any significant mineralization that exists in the fault block between the two deposits. If there is a mineable body in the Offset zone, it would increase the likelihood of Kemess East being exploited in the same manner as Kemess Underground.

Recent drilling at Kemess East confirmed that the Kemess East deposit remains open to the south and at depth. Further drilling is needed to test the mineral potential of this zone and fully define the extents of the deposit. Exploration potential exists, continuing east of the Kemess East deposit along the Kemess North mineralization trend, including the KEY and Hilda South brownfield exploration targets. These targets have a similar geophysical and geochemical signature as the other main deposits including KUG and Kemess East, consisting of shallow tabular chargeability high anomalies, with underlying chargeability lows that host the bulk of the mineralization. Step out drilling further east from the Kemess East deposit along trend is recommended.

The drilling data and updated resource model at Kemess South show potential for additional shallow and deep mineralization west of and below the historical pit. Future drilling is recommended west of the Kemess South open pit in the West Fault area to target potential resource expansion both at surface and at depth.

Additional greenfield exploration work is recommended to advance near surface porphyry targets in the Kemess district and adjacent properties. Recommended additional geophysics surveys include airborne mobile magnetotelluric (MT) surveys and detailed ground IP surveys, regional geochemistry surveys including stream sediment sampling as well as soil and till sampling grids, regional data compilation, and ongoing helicopter supported diamond drilling programs.

Budgets for programs over the next three years should be on par with Centerra programs in recent years. Annually, these budgets have been in the range of CA$4–10 million comprising diamond drilling (11,000–31,000 m), geophysics programs (15–50 line-km IP surveys and other surveys), geochemical

sampling, and development of a comprehensive 3D exploration model and drilling database that compiles lithology, alteration, mineralization, structural, geochemical, and geophysical information. Ongoing advancement of the 3D geology and exploration models for both the Kemess deposits and the Kemess greenfield district are recommended to improve understanding of the geometry of the deposit from fault block to fault block, to find extensions of known mineralization that could potentially be added to resources base, and to guide future discoveries.

Specific to the Main Zone and KUG zone, the QP Geologist makes the following recommendations:

- Complete additional infill and step-out drilling to improve definition of fault continuity, orientation, and spatial extent.
- Incorporate oriented core drilling and/or televiewer data, where feasible, to improve confidence in fault geometry and kinematic interpretations.
- Refine the structural model by integrating three-dimensional fault representations, where supported by data, to better constrain fault thickness and potential impacts on mineralization.
- Integrate and reconcile the CSA Global and in-house alteration models for the Kemess Main Zone.
- Expand spectral analysis and magnetic susceptibility data coverage to improve alteration classification.
- Develop and incorporate an alteration model for Kemess South.
- Update the geological and Mineral Resource models as additional drilling, structural, and alteration data become available.

26.2 GEOTECHNICAL ENGINEERING, HYDROGEOLOGY

Due to the interaction between the open pit and underground areas of the Main Zone, detailed design, scheduling and computational modelling is recommended. 3D computational modelling is recommended and should consider multiple milestones according to the mining schedule and projected advance of surface and underground mining.

- Laboratory testing of representative tailings material for suitability as cement paste backfill is recommended. This will be an important input for stope sizing, stope production scheduling and cost estimation.
- Additional phases of drilling from surface, associated laboratory testing and analyses are recommended to further characterize the rock mass of the Main Zone and Kemess Underground. Drilling should focus on locations for major infrastructure such as crushers,

underground excavations, tunnels, portal excavations, ventilation raises and development areas of the underground mine.

- Hydrogeological studies including permeability testing are recommended for the Main Zone. Instrumentation such as vibrating wire piezometers are also recommended in the Main Zone to monitor phreatic levels. Hydrogeological studies will provide important inputs for design of dewatering systems and operating and capital costs.
- Geohazard assessments of surface infrastructure and access roads are recommended to design appropriate engineering and administrative controls.
- Additional geotechnical and hydrogeological drilling, testing and appropriate studies are recommended for Kemess South prior to restarting mining activities.
- Additional geotechnical and hydrogeological engineering is projected to cost approximately US$5,000,000.

26.3 MINING

26.3.1 Geotechnical

Recommended pit slope geotechnical study should focus on the following key action items:

- Additional geomechanical and hydrogeological drilling, sampling and testing to cover the data gap zones of the proposed open pits
- Additional televiewer surveys in select exploration and/or geotechnical drillholes to enhance the rock mass structural database
- Upgrade of the 3D lithology, alteration, oxidization, and fault models, including the locations, continuities, and characteristics of local major structures
- Refinement of the geotechnical model, particularly for the broken zone delineation
- Development of a hydrogeology model for pit dewatering plan
- Enhancement of terrain hazard assessment to include ground truthing and refined stability analyses on the natural slopes surrounding the proposed pits.

26.3.2 Mining

Recommendations for mining pertain to advanced engineering studies towards completion of a prefeasibility study in 2026, including:

- Equipment trade-off studies, and updated manpower estimates. Trade-off studies should include utilization of new technologies such as automation and electrification.

- Collection of equipment price quotations, fabrication times and estimated delivery dates to provide updated capital costs and schedules.
- More detailed planning and scheduling to provide annual estimates of production, revenues and costs.
- Ventilation simulations with updated mine designs, equipment requirements and mining schedules to verify key design, capital and operating cost assumptions.
- Underground and surface material handling simulations to finalize mechanical designs, capital and operating cost projections.
- Evaluation of alternative locations for the Waste Rock Storage Facility.

26.3.3 Backfill System

Given the conceptual nature of the PEA design and the limited project-specific test work completed to date, additional studies are required to validate key assumptions and reduce technical uncertainty. To advance the paste backfill system beyond the PEA level and reduce technical uncertainty associated with conceptual assumptions, the following work is recommended in addition to future engineering study level progression (PFS, FS, etc.):

- Paste test work campaign – to define the unique characteristics of Kemess tailings and paste. This test work campaign would include material characterization, rheology testing, thickening and filtration testing and unconfined strength testing.
- Future studies:
  - Surface vs. Underground Paste Plant Location Trade-off Study – An underground paste plant may warrant evaluation due to challenging surface topography. An underground paste plant would potentially be located closer to the deposit which could potentially reduce or eliminate the need for an underground booster station.
  - Thickener Location Trade-off Study – The location of the thickener could be optimized to improve CAPEX and energy consumption of the overland tailings pump systems.
  - Binder Study – A binder study is warranted to confirm supply capacity due to the remote location of Kemess mine and the potentially high binder requirement of the backfill system.
  - Cyclone Study – The use of cyclones could be considered to develop a targeted PSD which may improve filtration and binder consumption.

26.4 METALLURGY AND MINERAL PROCESSING

The following recommendations outline the key technical actions needed to strengthen the metallurgical basis of the project and support the next phase of engineering. They focus on improving confidence in recovery projections, enhancing the understanding of variability of mineralized material, and ensuring the selected flowsheet and plant design are fully validated.

- Equipment price quotations/lead time scheduling, reagent estimation, power estimates, training cost estimation
- Complete variability work on SMC, Bond BWi, and Ai testing across Main Zone domains—especially Not-Broken material—to refine throughput, energy demand, and mill sizing assumptions
- Additional metallurgical testing is recommended to further characterize the variability within the deposit, especially at depth and in different alteration zones.
- Conduct more leach testing across mine-life composites, investigate cyanide-soluble copper impacts, and refine the 70% gold leach recovery assumption with broader datasets.
- Undertake more detailed gold liberation and deportment studies, ensuring consistent grind-size reporting and sample naming across all mineralogical datasets.
- Complete detailed inspections of the existing equipment in the mill to de-risk future engineering works in the process plant.
- Optimize the flotation circuit by completing trade off studies for flotation technology and overall circuit layout.

26.5 INFRASTRUCTURE

Trade-off studies are recommended to optimize infrastructure locations, including the WRSF. Consideration should be made to minimize exposure to geological and other hazards. Additionally, infrastructure siting should consider weather conditions, serviceability and constructability.

Quotations for critical infrastructure, first-fills and critical components should be requested from reputable manufacturers.

26.5.1 Tailings Infrastructure

The following activities are recommended as part of the ongoing design of the KUG and KS TSFs:

- Conduct testwork to establish composition and strength parameter of tailings. This study has been based on tailings testwork completed as part of the Kemess Underground development

(Golder, 2019) which may vary from the Nugget and Main Dam Pit tailings. Testwork should include:

  – Index testing
  – Settling and consolidation analysis
  – Chemical testing to confirm rougher tailings assessment of whether they are acid-generating and/or metal leaching
  – Material strength testing, specific proposed cyclone sand material strength testing.

- Site investigation, detailed characterization, and sub-surface material interpretation for the diversion dams and North Saddle Dam are required to refine the design. Currently, minimal data is available in these locations, and with updated designs larger than the initial footprints, additional areas need to be investigated.
- Detailed stability and sensitivity assessments are required to evaluate dam structural performance using refined input parameters based on confirmed embankment materials and laboratory testing. Detailed seepage analyses and modelling should be conducted for all embankments utilizing updated material properties to estimate the phreatic surface and evaluate potential impacts on downstream water management infrastructure.
- More detailed design of access roads, diversions, and closure spillway.
- Complete a water balance model that reflect the tailings management strategy within both facilities.
- An updated deposition model should be developed, incorporating revised pond sizing and a defined tailings surface. This should consider minimum tailings beach length of 500 m, a minimum of 2 m water depth below the barge for adequate water reclaim, and maintenance of the pond in the optimal location within the facility.
- Assessment of the existing water management structures including the KS TSF Seepage Recycling Pond and Sediment Pond to determine current capacity and if they accommodate TSF raises.
- Establish the cyclone sand placement methodology.
- Review existing site investigation data and perform additional site investigation to characterize foundation conditions, material strength parameters, specifically under the SDD, NDD, proposed EDD, purposed North Dam area, roads, spillways where limited data exists.
- Further optimization of diversion and water management design and associated diversion structures.

- Further definition of tailings geotechnical and geochemical characteristics.
- Detail tailings deposition to define staged embankment construction and staged tailings deposition.
- Updated stability assessments incorporating refined input parameters for cyclone sand, based on further review of existing material properties and finalized tailings splits.
- Preliminary engineering designs for supporting infrastructure, including, access roads, tailings pipelines, cyclone facilities, water management ponds and systems.
- Water quality modelling to support closure plans and project cost.

26.6 ENVIRONMENTAL AND SOCIAL

Recommended tasks for environmental stewardship and social interactions include:

- Additional studies to assess new areas to be disturbed such as: new mine waste storage facilities, the conversion of the existing permitted block cave subsidence zone to an open pit configuration, new conveyor alignment, infrastructure, etc. This will include, but not be limited to, additional studies for geochemical caharzterization and water quality and quantity, fisheries and aquatic resources, archaeology, soils, vegetation, and wildlife.
- Continue and finalize field work to complete the environmental studies commenced in 2024. with the aim to have baseline collection completed by the end of 2027/early 2028.
- Complete a Site Performance Objective (SPO) to establish long-term selenium permit limits for Waste Rock Creek, supported by ongoing studies such as the Bird and Amphibian Selenium Bioaccumulation study initiated in early 2025 to manage selenium inputs from a previously operated WRSF in the Kemess South Mine Area.
- Continue discussions with affected First Nations to maintain Project transparency and support and consider potential addendums to the IBA.

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28 GLOSSARY OF UNITS, ABBREVIATIONS, AND SYMBOLS

28.1 GLOSSARY OF UNITS

Unit	Definition
'	foot/feet
"	inches
#	number
%	percent
/	per
‘	minutes (geographic)
”	seconds (geographic)
<	less than
>	greater than
°	degrees
°C	degrees Celsius
µm	micrometres (microns)
a	annum (year)
Å	angstroms
asl	above sea level
BQ	36.5 mm diameter core
c.	circa
d	day
fineness	parts per thousand of gold in an alloy
ft	feet
g	gram
g/cm^3	grams per cubic centimetre
g/m^3	grams per cubic metre
Ga	billion years ago
ha	hectares
HP	horsepower
HQ	63.5-mm diameter core
kg/m^3	kilograms per cubic metre
kL	kilolitres
km	kilometre
km^2	square kilometres
koz	thousand ounces
kton	thousand tonnes
kV	kilovolt
kVA	kilovolt–ampere
kW	kilowatt
kWh	kilowatt hour
kWh/t	kilowatt hours per tonne
lb	pound

Unit	Definition
M	million(s)
m	metres
m^3	cubic metres
m^3/h	cubic metres per hour
Ma	million years ago
Mesh	size based on the number of openings in one inch of screen
mg/L	milligrams per litre
mi	mile/miles
Mlb	million pounds
Mm	million metres
mm	millimetres
Mm^3	million cubic metres
Moz	million ounces
mRL	metres relative level
Mt	million tonnes
Mtpa	million tonnes per annum
MW	megawatts
NQ/NQ2	47.6 mm size core
oz	ounce/ounces (Troy ounce)
P	Passing, i.e. % passing through a screen
pH	measure of the acidity or alkalinity of a solution
pop	population
ppb	parts per billion
ppm	parts per million
PQ	85 mm diameter core
t	metric tonne(s)
t/m^3	tonnes per cubic metre
TDS	total dissolved solids
tpa	tonnes per annum (tonnes per year)
tpd	tonnes per day
tph	tonnes per hour
tpod	tonnes per operating day
TSS	total suspended solids
wt%	weight percent

28.2 GLOSSARY OF ABBREVIATIONS

Abbreviation	Definition
®	registered name
AAS	atomic absorption spectroscopy
AC	aircore
ADR	adsorption, desorption and recovery
AISC	all-in sustaining cost(s)
Amdel	Amdel Laboratory
Anaconda	Anaconda Canada Ltd
ANC	acid-neutralizing capacity
ANP	acid-neutralizing potential
ARD	acid-rock drainage
AuAA	cyanide-soluble gold
AuEq	gold equivalent
AuFA	fire assay
AuPR	preg-rob gold
AuRico	AuRico Metals Inc.
AuSF	screen fire assay
BFA	bench face angle
BLEG	bulk leach extractable gold
BMCO	breakeven mill cut-off
BSCO	breakeven stockpile cut-off
C.P.G.	Certified Professional Geologist
CA	Canadian
CA$	Canadian Dollar
CAF	cost adjustment factor
capex	capital expenditure
CDA	Canadian Dam Association
Centerra	Centerra Gold Inc.
CER	Consultative Environmental Review
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CN_{wad}	weak acid-dissociable cyanide
COS	coarse ore stockpile
CRF	capital recovery factor
CRM	certified reference material
CST	cleaner scavenger tailings
CTOT	Total carbon
Cu Eq	copper equivalent
CuCN	cyanide-soluble copper
DCF	discounted cashflow
E	east
EAC	Environmental Assessment Certificate

Abbreviation	Definition
EDA	exploratory data analysis
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
El Condor	El Condor Resources Ltd
EMA	Environmental Management Act
EMC	Environmental Management Committee
ENV	Ministry of Environment and Parks
EOM	end of month
EOY	end of year
EPA	Environmental Protection Authority
ERMP	Environmental Review and Management Program
FS	feasibility study
G&A	general and administration
GAAP	Generally Accepted Accounting Principles
GN	mine grid north
GPS	global positioning system
GRG	gravity recovery gold
H	horizontal
HC	high capacity
HPGR	high pressure grinding rolls
HSRC	Health, Safety and Reclamation Code
IAAC	Impact Assessment Agency of Canada
IBA	Impact Benefit Agreement
ICP	inductively-coupled plasma
ICP-AES	inductively-coupled plasma atomic emission spectroscopy
ICP-MS	inductively-coupled plasma mass spectrometry
ICP-OES	inductively-coupled plasma optical emission spectrometry
IP	induced polarization
IRA	inter-ramp slope angle
IT	information technology
IW	Impacted Water
JCR	joint condition rating
JV	joint venture
KOZ	Kemess Offset Zone
KP	Knight Piésold
KUG	Kemess Underground
KV	kriging variance
LC	low capacity
L–G	Lerchs–Grossmann
LOA	length overall
LOM	life-of-mine
LSK	large-scale kinetic
MCM	Ministry of Mines and Critical Minerals
MIK	multiple-indicator kriging

Abbreviation	Definition
MMSA	Mining and Metallurgical Society of America
MN	magnetic north
MPA	maximum potential acidity
MRF	Mine Rehabilitation Fund
MSE	mean squared error
MWMS	mine water management system
MWMT	meteoric water mobility testing
N	north
NAG	net acid generation/net acid generating
NAPP	net acid-producing potential
NI 43-101	Canadian National Instrument 43-101 “Standards of Disclosure for Mineral Projects”
NN	nearest-neighbor
NNP	net neutralizing potential
NOI	Notice of Intent
Northgate	Northgate Exploration Ltd
NPV	net present value
NSR	net smelter return
NW	northwest
OK	ordinary kriging
opex	operating expenditure
OSA	overall slope angle
P.Geo.	Professional Geologist (CAN)
P.E.	Professional Engineer (US)
P.Eng.	Professional Engineer (CAN)
P.G.	Professional Geologist (US)
Pacific Ridge	Pacific Ridge Resources Ltd
PAG	potentially acid-generating
PFS	pre-feasibility study
PLI	point load index
PSD	particle size distribution
PSI	pounds per square inch
pXRF	portable x-ray fluorescence
QA/QC	quality assurance and quality control
QLT	quick leach test
QP	Qualified Person
RAB	rotary air blast
RC	reverse circulation
RDA	Residue Disposal Area
RF	revenue factor
Rio Algom	Rio Algom Explorations Inc.
RMR	rock mass rating
ROM	run-of-mine
Royal Oak	Royal Oak Mines Inc.
RPL	Environmental Monitoring Plan

Abbreviation	Definition
RQD	rock quality designation
S	south
S&ER	Sustainability and External Relations
SAG	semi-autogenous grind
SE	southeast
SEIS	Supplemental Environmental Impact Statement
SG	specific gravity
SMU	selective mining unit
SRM	standard reference material
SS	sulfide sulphur
ST	scavenger tailings
St. Philips	St. Philips Resources Inc.
STOT	Total sulphur
SWIR	short-wave infrared
SX-EW	solvent extraction–electrowinning
TF	tonnage factor
TN	true north
Topo	topography
TSF	tailings storage facility
UC	uniform conditioning
UG	underground
US	United States
US$	United States dollar(s)
V	vertical
VWP	vibrating wire piezometer
W	west
WD	waste dump
WDX	waste dump expansion
WRSF	waste rock storage facility

28.3 GLOSSARY OF SYMBOLS

Symbol	Element
Ag	silver
Al	aluminum
As	arsenic
Au	gold
B	boron
Ba	barium
Be	beryllium
Bi	bismuth
C	carbon
Ca	calcium
$CaCO_3$	calcium carbonate
CaO	calcium oxide

Symbol	Element
$CaSO_4•2H_2O$	calcium sulphide dihydrate
Cd	cadmium
Ce	cerium
Cl	chlorine
CN	cyanide
CO	carbon monoxide
Co	cobalt
Cr	chromium
Cs	caesium
Cu	copper
Fe	iron
FeOx	iron oxides
Ga	gallium
Ge	germanium
H	hydrogen
Hf	hafnium
Hg	mercury
In	indium
K	potassium
La	lanthanum
Li	lithium
Mg	magnesium
Mn	manganese
$Mn(OH)_2$	manganous hydroxide
MnO_2	manganese dioxide
Mo	molybdenum
N	nitrogen
Na	sodium
Nb	niobium
NH_3	ammonia
Ni	nickel
NOx	nitrogen oxide compounds
O_2	oxygen
P	phosphorus
Pb	lead
Pd	palladium
Pt	platinum
Rb	rubidium
Re	rhenium
S	sulphur
Sb	antimony
Sc	scandium
Se	selenium