

# KATANNING GOLD PROJECT

## OPEN PIT DEFINITIVE FEASIBILITY STUDY

Client: Ausgold Ltd

Issue Date: 24/6/2025

Status: Final

Doc. No.: Rev 0.2

Document Identification / History / Status / Distribution

Identification

Report #: 1

Last Saved: 24 June 2025

File Name & Path:

P:\Current Projects\1092-Ausgold-Kattanningdfs\04\_Deliverables\01\_Report\02\_Final\1092\_Ausgold\_Katanning Gold Project\_DFS\_Report\_Rev0p2\_20250624.Docx

Internal Review

Category	Rev. #	Author	Date	Reviewed by	Comments
Draft	B	JJF	31/3/25	A.Hollis	Internal Peer Review

External Review and Distribution

Category	Rev. #	Ver. #	Issued By	Issued to	Date	Comments
Draft	D	1	JJF	R. McAuslane	1/4/2025	First Draft for Client review
Draft	E	1	JJF	R. McAuslane	14/4/2025	Revised Contractor rates
Final	0	0	JJF	R. McAuslane	12/5/2025	Removed TSF costs captured by others.
Final	0	1	JJF	R. McAuslane	16/5/2025	Typo correction
Final	0	2	JJF	R. McAuslane	16/5/2025	Change to 3 month pre-production

---

## TABLE OF CONTENTS

---

<b>1</b>	<b>EXECUTIVE SUMMARY</b>	<b>14</b>
1.1	ORE RESERVE SUMMARY	14
1.2	MINING METHOD	15
1.3	MINING FACTORS	15
1.4	PIT OPTIMISATION	16
1.5	MINE DESIGN	20
1.6	MINE PRODUCTION SCHEDULE	25
1.7	MINE COST ESTIMATION	29
1.8	FINANCIALS	30
1.9	RISKS AND OPPORTUNITIES	31
1.9.1	Execution Risks	31
1.9.2	Mining Cost Risks	32
1.9.3	Execution Opportunities	32
<b>2</b>	<b>INTRODUCTION</b>	<b>33</b>
2.1	BACKGROUND	33
2.2	LOCATION	33
2.3	GEOLOGY	34
2.4	HISTORICAL MINING	38
2.5	TENEMENT BOUNDARIES	38
2.6	TOPOGRAPHY	38
2.7	CLIMATE	41
2.8	CONVENTIONS	41
2.8.1	Currency	41
2.8.2	Units	41
2.8.3	Mine Coordinates	42
2.8.4	Block Dimensions	42
2.9	SCOPE OF WORK	42
2.9.1	General	42
2.9.2	Deliverables	42
2.9.3	Exclusions	43
<b>3</b>	<b>MINING CONSIDERATIONS</b>	<b>44</b>
3.1	RESOURCE MODEL	44
3.2	DILUTION AND ORE LOSS	46
3.3	WATER MANAGEMENT	50

3.3.1	Surface Water	50
3.3.2	Ground Water	50
3.4	GEOTECHNICAL	51
3.4.1	Rock Properties	51
3.4.2	Faulting	51
3.4.3	Seismicity	52
3.4.4	Slope Design	54
<b>4</b>	<b>MINING METHODOLOGY</b>	<b>55</b>
4.1	GENERAL	55
4.2	MINING METHOD	55
4.3	MINE INFRASTRUCTURE	56
4.4	CLEARING & TOPSOIL REMOVAL	56
4.5	GRADE CONTROL	58
4.6	DRILL AND BLAST	58
4.7	LOAD AND HAUL	59
4.8	ANCILLARY SERVICES	59
4.9	DEWATERING	60
4.10	ROM MANAGEMENT	60
<b>5</b>	<b>OPEN PIT OPTIMISATION</b>	<b>61</b>
5.1	OPEN PIT OPTIMISATION OVERVIEW	61
5.2	OPEN PIT OPTIMISATION METHODOLOGY	61
5.3	INPUTS AND ASSUMPTIONS	63
5.3.1	Gold Price	63
5.3.2	Selling Costs	63
5.3.3	Wall Slope Parameters	63
5.3.4	Mining Costs	64
5.3.5	Processing Inputs	65
5.3.6	Process Recoveries & Breakeven cut-off grade	66
5.3.7	Capital Costs Inputs	67
5.4	OPTIMISATION RESULTS	67
5.4.1	Central Zone	67
5.4.2	South Zone	69
5.4.3	North Zone	70
5.4.4	Shell Selection	72
<b>6</b>	<b>MINE DESIGN</b>	<b>78</b>
6.1	MINE DESIGN CRITERIA	78

6.1.1	Design Objective	78
6.1.2	Open Pit Slopes	78
6.1.3	Waste dump slopes	79
6.1.4	Ramps	79
6.1.5	Minimum Mining Width	80
6.2	OPEN PIT MINE DESIGN PHYSICALS	81
6.2.1	General	81
6.2.2	Jinkas Mine Area	81
6.2.3	Jackson and Olympia Mine Area	85
6.2.4	Dingo Pit Design	88
6.3	STAGE INVENTORIES	91
6.4	WASTE DUMP DESIGN	92
6.5	GENERAL SITE LAYOUT	94
6.5.1	ROM Design	96
6.5.2	Warren Road Diversion Strategy	97
<b>7</b>	<b>MINE PRODUCTION SCHEDULE</b>	<b>100</b>
7.1	SCHEDULING METHOD	100
7.2	SCHEDULING OBJECTIVES AND CONSTRAINTS	101
7.2.1	Scheduling Periods	101
7.2.2	Stage Release	102
7.2.3	Workdays per Year	102
7.2.4	Shifts and Shift Duration	102
7.3	PRODUCTION SCHEDULE ASSUMPTIONS	102
7.3.1	Mining Targets and Constraints	102
7.3.2	Processing	103
7.3.3	Mining Productivity	104
7.3.4	Fleet Levels	105
7.3.5	Stockpiling Strategy	105
7.3.6	Warren Road Diversion	105
7.4	PRODUCTION SCHEDULE	105
7.4.1	Total Material Mined	108
7.4.2	Ore Mined and Processed	108
<b>8</b>	<b>MINE OPERATIONS AND MANAGEMENT</b>	<b>126</b>
8.1	MINE OPERATIONS APPROACH	126
8.2	ROSTER	126
8.3	MINE MANAGEMENT & TECHNICAL SERVICES	127

8.4	MANNING LEVELS	127
<b>9</b>	<b>INFRASTRUCTURE AND SERVICES</b>	<b>128</b>
9.1	MINE SERVICES AREA	128
9.2	SITE PREPARATION	130
9.2.1	Pre-Production Dewatering	130
9.2.2	Haul Roads	130
9.2.3	Clear and Grub	130
<b>10</b>	<b>MINING COST ESTIMATES</b>	<b>132</b>
10.1	BASIS OF ESTIMATE	132
10.2	CONTRACTOR SUBMISSIONS	133
10.3	CAPITAL COSTS	134
10.3.1	Owners Equipment	134
10.3.2	Contractor Establishment	135
10.3.3	Mobilisation	135
10.3.4	Pre-Production Costs	136
10.4	OPERATING COSTS	137
10.4.1	Owner Costs	137
10.4.2	Contractors Overheads	138
10.4.3	Mine Development	138
10.4.4	Grade Control	138
10.4.5	Drill and Blast	140
10.4.6	Load and haul	140
10.4.7	Ore Rehandle	144
10.4.8	Rehabilitation	144
10.4.9	Fuel Consumption	144
10.5	MINING COST SUMMARY	145
<b>11</b>	<b>FINANCIALS</b>	<b>146</b>
<b>12</b>	<b>ORE RESERVE</b>	<b>148</b>
12.1	SITE VISIT	148
12.2	COMPETENT PERSON	148
12.3	ORE RESERVE	149
<b>13</b>	<b>RISKS AND OPPORTUNITES</b>	<b>150</b>
13.1	EXECUTION RISKS	150
13.2	MINING COST RISKS	150
13.3	EXECUTION OPPORTUNITIES	150
<b>14</b>	<b>BIBLIOGRAPHY</b>	<b>152</b>

---

## LIST OF FIGURES

---

Figure 1-1: Pit Optimisation Sensitivity Runs (showing changes with $\pm 10\%$ variance in inputs)	18
Figure 1-2: Scenario 103 Pit Optimisation Results for CZ	19
Figure 1-3 Scenario 206 Pit Optimisation Results for SZ	20
Figure 1-4: Design Stages	23
Figure 1-5: Waste Rock Dump Design and Overall Layout	24
Figure 1-6: Annualised Schedule Results	28
Figure 1-7: Annual AISC per ounce	31
Figure 1-8: Cumulative Discounted Cashflow	31
Figure 2-1: Katanning Gold Project Location	33
Figure 2-2 Access to the KGP Site	34
Figure 2-3 Isometric view of local geology looking from northeast to southwest	35
Figure 2-4: Cross-section through geology – Jackson Pit	36
Figure 2-5: Cross-section through geology – Jinkas Main Pit	37
Figure 2-6: Cross-section through geology – Dingo Pit	37
Figure 2-7 Western Wall of Jinkas Pit wall showing well developed foliation	38
Figure 2-8 KGP Topography in 10m contours	39
Figure 2-9 Aerial view of KGP Site with approximate pit location	40
Figure 3-1 Initial Model Cross Section through Jackson & Olympia	44
Figure 3-2 Initial Model Cross Section through Jinkas South	45
Figure 3-3 Regularisation Process to a Parcel Model	47
Figure 3-4 Determination of Zone of Mixing	47
Figure 3-5 Swapping of material within Mixing Zone	48
Figure 3-6 Grade Tonnage curves for Undiluted and Diluted options	49
Figure 3-7: Current known Regional fault	52
Figure 3-8: NSHA23 hazard map indicating the mean PGA for 10% probability of exceedance in 50-years	53
Figure 4-1: Proposed Site Layout	57
Figure 5-1: Worst Case DCF Mining Sequence	63
Figure 5-2: Best Case DCF Mining Sequence	63
Figure 5-3: Slope Design Geometrics	64
Figure 5-4: Mining Costs by Level	65
Figure 5-5 Recovery Curves	67

Figure 5-6 Waterfall charts for Optimisation Runs Focusing on Sensitivity to Constraints	68
Figure 5-7: Central Zone Pit Optimisation Sensitivity Runs (showing changes with $\pm 10\%$ variance in inputs)	69
Figure 5-8 Scenario 103 Central Zone Base Case Optimisation	76
Figure 5-9 Comparison of Base case and Selected shells for CZ	76
Figure 5-10 Scenario 206 South Zone Base Case Optimisation	77
Figure 5-11 Comparison of Base case and Selected shells for SZ	77
Figure 6-1 As-built and final rehabilitated slope dimensions	79
Figure 6-2 Dual Lane Ramp Layout	80
Figure 6-3 Single Lane Ramp Layout	80
Figure 6-4 Minimum Mining Width	81
Figure 6-5 Pit Design – Jinkas Mine Area	83
Figure 6-6 Jinkas Designs versus Optimisation Shell – Plan and Sections	84
Figure 6-7 Pit Design – Jackson and Olympia Mine Area	86
Figure 6-8: Jackson and Olympia Designs versus Optimisation Shell – Plan and Sections	87
Figure 6-9 Pit Design – Dingo Mine Area	89
Figure 6-10 Dingo Designs versus Optimisation Shell – Plan and Sections	90
Figure 6-11 Ultimate Pit and Waste Dump – Central Layout	92
Figure 6-12 Ultimate Pit and Waste Dump – South Zone Layout	93
Figure 6-13: Proposed Layout and Location	95
Figure 6-14 ROM Pad Design	96
Figure 6-15 Location of Traffic control during Phase 1 and Phase 2	98
Figure 6-16 Phase 3 Warren Road Diversion 1	98
Figure 6-17 Phase 4 Final Warren Road Diversion	99
Figure 7-1 Pit Stage Layout for LOM Production Schedule	102
Figure 7-2 KGP Mining Schedule Mining Sequence	109
Figure 7-3 KGP Mining Schedule Bench Turnover	109
Figure 7-4: KGP Mining Schedule Ex-pit mined by Stage	110
Figure 7-5 KGP Mining Schedule Ore Mining Ex-pit by Stage	111
Figure 7-6 Stockpile Balance	111
Figure 7-7 KGP Mining Schedule Crusher Feed	112
Figure 7-8 Mining Schedule Plant Feed Hours	113
Figure 7-9 Gold Production and Recovery	114
Figure 7-10: End of period plans – pre-production	115

Figure 7-11: End of period plans – 1 <sup>st</sup> year	116
Figure 7-12: End of period plans – 2 <sup>nd</sup> year	117
Figure 7-13: End of period plans – 3 <sup>rd</sup> year	118
Figure 7-14: End of period plans – 4 <sup>th</sup> year	119
Figure 7-15: End of period plans – 5 <sup>th</sup> year	120
Figure 7-16: End of period plans – 6 <sup>th</sup> year	121
Figure 7-17: End of period plans – 7 <sup>th</sup> year	122
Figure 7-18: End of period plans – 8 <sup>th</sup> year	123
Figure 7-19: End of period plans – 9 <sup>th</sup> year	124
Figure 7-20: End of period plans – 10 <sup>th</sup> year (Ultimate pit and waste rock dump)	125
Figure 8-1 Organisational Structure for the Owners Mining Team	127
Figure 9-1 Mine Service Area Layout	129
Figure 9-2: Pit and Waste Dump Extents (together with area needed for clearing)	131
Figure 11-1: ASIC per ounce	146
Figure 11-2: Cumulative Cashflow	146
Figure 12-1 Remaining Core after Testing	148

---

## LIST OF TABLES

---

Table 1-1: Katanning Gold Project Open Pit Ore Reserve Summary	15
Table 1-2: Resource Inventory by Category	16
Table 1-3: Pit Optimisation Parameters	17
Table 1-4: Pit Geotechnical Parameters for Pit Optimisation	18
Table 1-5 Detailed Results for Selected Shells	19
Table 1-6: Inventories by Stage	21
Table 1-7 Design to Pit Shell Reconciliation	22
Table 1-8: KGP Annual Mining Schedule	27
Table 1-9 Annual Mining Cost Summary by Cost Centre	29
Table 1-10: Open Pit Project Financials	30
Table 2-1: Average Temperature Data for Katanning (source: <a href="http://www.bom.gov.au">www.bom.gov.au</a> )	41
Table 3-1: Resource Model Characteristics	45
Table 3-2: Resource Tonnage by Model and Resource Category where Au $\geq$ 0.45g/t, above 150mRL	46
Table 3-3 Apparent Mixing Widths	48
Table 3-4 CZ Dilution modelling sensitivity	49

Table 3-5: Dilution Modelling Results	50
Table 3-6 Rock Mass Properties	51
Table 3-7: Open Pit Design Slope Design Parameters	54
Table 4-1: Drill and Blast Design Parameters	59
Table 5-1: Geotechnical Parameters for Pit Optimisation and Mine Design	64
Table 5-2: Pit Optimisation Parameters	65
Table 5-3: Process unit costs	66
Table 5-4 Tailing Grade Based Recovery	66
Table 5-5: Central Zone Pit Optimisation Runs Focusing on Sensitivity to Various Parameters	68
Table 5-6 South and North Zone Pit Optimisation Runs Focusing on constraints	71
Table 5-7 Summary of Selected Shell Inventories	73
Table 5-8: Scenario 103 Central Zone Results (showing highest, lowest and average discounted cashflows with selected shell highlighted in pink)	74
Table 5-9 Scenario 206 South Zone Results (showing highest / lowest discounted cashflows with selected shell highlighted in pink)	75
Table 6-1 Revised Open Pit Slope Design Parameters	78
Table 6-2: Waste Dump Design Criteria	79
Table 6-3: Designed Ramp Width's	80
Table 6-4 Jinkas Shell to Design Comparison	82
Table 6-5 Jackson & Olympia Shell to Design Comparison	85
Table 6-6 Dingo Shell to Design Comparison	88
Table 6-7: Inventories by Option and by Stage	91
Table 6-8 Inferred within Pits	91
Table 6-9: Waste dump and Backfill Statistics	94
Table 7-1 Scheduling Periods	101
Table 7-2 Throughput rate (tph)	103
Table 7-3 Plant Ramp up	104
Table 7-4 Loading Productivity Assumptions	104
Table 7-5 Mining Rate by Period	105
Table 7-6 Stage Ranking	106
Table 7-7: KGP Mining Schedule	107
Table 9-1: Clearing Area & Topsoil Volumes	130
Table 10-1 Contractor Submissions	133
Table 10-2 Owner Equipment	135

Table 10-3 Contractor Site Establishment	135
Table 10-4 Contractor Equipment Mobilisation	136
Table 10-5 Pre-Production Costs	136
Table 10-6 Owners Mining Department Personnel	137
Table 10-7 General Overheads and Consumables	137
Table 10-8 Contractor Monthly Management Fees & Overheads	138
Table 10-9 Mine Redevelopment Unit Rates	138
Table 10-10 Grade Control Drilling quantities (including Pre-production)	139
Table 10-11 Grade Control Assay quantities (including Pre-production)	139
Table 10-12 Drill & Blast Unit Rates	140
Table 10-13 Ore haulage unit rate to ROM pad (\$/bcm)	141
Table 10-14 Ore haulage unit rate to Stockpile (\$/bcm)	142
Table 10-15 Waste haulage unit rate to WRD (\$/bcm)	143
Table 10-16 Rehandle Unit Rates	144
Table 10-17 Rehabilitation Unit Rates	144
Table 10-18 Fuel Consumption	144
Table 10-19 Annual Mining Operating Cost Summary by Cost Centre	145
Table 11-1: EBIT Level Cashflow	147
Table 12-1: Ore Reserve	149

## Glossary of Acronyms/Abbreviations

AUC	Ausgold Limited
BCM	Bank Cubic Metres (i.e. In-situ volume)
CIL	Carbon in leach processing method
COG	Break-even Cut-off Grade - Grade above which mineralisation is reported
csv	comma separated values
DCF	Discounted Cash Flow
DXF	digital exchange format for drawing files
FEL	Front end Loader
g/t	grams per tonne
G&A	General and Administrative
JORC	Joint Ore Reserves Committee (Australian reporting standards for mineral projects)
KGP	Katanning Gold Project
km	Kilometre
koz	thousand ounces
LOM	Life of Mine
m	Metres
Mbcm	Million Bank Cubic Metres
Mlcm	Million Loose Cubic Metres
MPSO	Mineplan Schedule Optimiser
mRL	Metres Relative Level
Mt	Million tonnes
Mtpa	Million tonnes per annum
NPV	Net Present Value
OMP	Ore Mining Premium
Oz	Troy ounce
PFS	Preliminary Feasibility Study
ppm	parts per millions (equivalent to grams per tonne)
QA/QC	Quality Assurance / Quality Control
RC	Reverse Circulation
ROM	Run of Mine (referring to un-processed ex-pit ore materials)
SR	Strip Ratio (i.e. waste/ore)
SSE	Senior Site Executive
t	Tonne (metric)
tph	tonnes per hour
TSF	Tailings Storage Facility
TMM	Total Material Movement
TRW	Total Ramp Width
WHITTLE <sup>™</sup>	Geovia Strategic Mine Planning Software
WRD	Waste Rock Dump

## **Disclaimer**

### Confidentiality

This document is confidential and may not be disclosed, reported, copied, quoted, or published without the written consent of both Orelogy Mine Consultants Pty Ltd (Orelogy) and Ausgold Ltd.

### Liability

Orelogy accepts no liability for any loss or damage arising because of any person / group other than Ausgold Ltd acting in reliance on any information, opinion or advice provided in this document.

### Reliance

This document may not be relied upon by any person other than Ausgold Ltd, its officers and employees.

### Information

Orelogy accepts no liability and gives no warranty as to the accuracy or completeness of information provided by or on behalf of Ausgold Ltd, its representatives or other third party groups and takes no account of matters that existed when the document was delivered to Ausgold Ltd but which were not known to Orelogy at this time.

### Recommendations

Orelogy accepts no liability for any matters arising if any recommendations contained within this document are not carried out, or are partially carried out, without further advice being obtained from Orelogy.

### Public Reporting

If Ausgold Ltd wishes to publish a Mineral Resource or Ore / Mineral Reserve estimate that is contained within this document, it must first obtain the relevant Competent / Qualified Person's written consent, not only to the estimate being published but also to the form and context of the published statement. The published statement must include a statement that responsible person's or Competent / Qualified Person's written consent has been obtained.

### Independence

Orelogy has no beneficial interest in the outcome of this technical study.

# 1 EXECUTIVE SUMMARY

Ausgold Limited (AUC) is a Perth-based gold exploration and development company. Its primary focus is the 100%-owned Katanning Gold Project (KGP), which covers >4,000 km<sup>2</sup> of the Katanning greenstone belt in south-western Western Australia. AUC is planning to develop a 3.6 Mtpa processing facility at site. Ore will be processed using single stage crushing, grinding and conventional gold processing Carbon in Leach (CIL) technology to produce gold bullion which will be sold to a nearby gold refinery for further processing. The process plant will also produce bullion via gravity gold concentration, intensive cyanidation and electrowinning.

The objective of this Study was to determine a practical and robust economic open pit mine plan that:

- Maximised the recovery of the current mineral resource.
- Delivered continuous ore feed to the process plant for a minimum 10 year mine life.
- Determined the mining cost for a Class 3 estimate.
- Identify the risks and opportunities.

This study has identified a technically and economically viable open pit that can fully utilise the design processing capacity for the ten year mine life after and 4 month preproduction phase.

This preferred go-forward case is based on conventional open cut operation using rigid dump trucks and backhoe excavators mining up to 35 Mtpa total material.

The proposed plan develops a total of 18 pits/stages to minimise waste stripping and provide access to lower strip ore as soon as possible. Up to four mining fleets will be utilised for operational flexibility and to ensure multiple dig faces are open at any given point in time.

## 1.1 Ore Reserve Summary

Orelogy has developed the open pit Ore Reserve in accordance with the Australian Joint Ore Reserve Committee (JORC 2012) guidelines.

The reported Mineral Resource estimate is inclusive of the resources converted to Ore Reserves. The Katanning Gold Project Feasibility Study Ore Reserve, as outlined in Table 1-1, was reported using variable cut-off grades, as summarised in Table 1-3. The cut-off grades vary by mineral type due to variations in process recoveries and costs.

Table 1-1: Katanning Gold Project Open Pit Ore Reserve Summary

	Category	Ore (Mt)	Au Grade (g/t)	Contained Gold (koz)
Central Zone	Proved	27.6	1.15	1,015.3
	Probable	4.8	0.96	146.9
	Sub-total	32.3	1.12	1,162.2
South Zone	Proved	1.2	0.97	36.5
	Probable	1.7	1.01	54.6
	Sub-total	2.9	0.99	91.0
<b>TOTAL</b>		<b>35.2</b>	<b>1.11</b>	<b>1,253.2</b>

The Competent Persons responsible for the Reserve is Jake Fitzsimons. Hossein Parhoon, a member of the study team visited the site in October 2024.

## 1.2 Mining Method

The operation is based on conventional hard rock mining using standard drill/blast/load/haul mining cycles. A mining contractor will be appointed to develop the open pit and will be responsible for all aspects of the operation. The Principal will be responsible for all technical and managerial duties.

Mining will utilise a staged approach with commencement of a footwall mining stage (west wall of Jinkas pit), followed by other low strip stages. The final hanging wall (north wall) stage is mined to a final depth of 220 m below average topography. The mine plan has been developed based on using 250 t class excavators and 140 t class rigid dump trucks for waste haulage. A smaller 120 t class excavator and 90 t class trucks will be used for selective mining of ore to minimise ore loss and dilution.

## 1.3 Mining Factors

The resource model contains no allowance for dilution or mining recovery except for internal dilution added during the sample compositing process. As part of the development of a credible and robust mine plan, dilution and mining recovery were considered. A diluted mining model was developed for the shallow dipping narrow vein orebody averaging 25% ore loss and 26% dilution.

Except for loose material at surface, all insitu material will require drill and blasting.

During the pre-production period, waste will be used for construction of the ROM pad, TSF embankment and other infrastructure, after which all waste will be hauled to the nearest waste dump or pit allocated for backfill.

## 1.4 Pit Optimisation

The Whittle™ open pit optimisation software tool was utilised by Orelogy to undertake this component of the study. Whittle™ is a recognised optimisation package and is considered the industry standard for open pit optimisation.

The strategy for the project is to maximise value for 3.6 Mtpa processing plant with a mine life of at least 9 years, which requires a shell with at least 32.4 Mt of ore.

Table 1-3 summaries all the major parameters used as part of the pit optimisation process.

Only material in the Measured and Indicated resource categories were included in the final pit optimisation runs for mine design purposes.

All blocks in the model were depleted based on the current as-mined topography, with a final inventory shown in Table 1-2.

Table 1-2: Resource Inventory by Category

Resource Category	Potential Ore $\geq$ 0.45 Au ppm	
	Mt	Au ppm
Measured	36.2	1.29
Indicated	27.8	1.06
Inferred	16.9	1.04
<b>Total</b>	<b>80.9</b>	<b>1.16</b>

At the commencement of the mining study in Sept 2024, the gold price was set at A\$3,000/oz.

A royalty rate of 2.5% was applied across the open pit optimisation work together with refining cost of \$7.50/oz.

Table 1-3: Pit Optimisation Parameters

Item	Description	Units	OXIDE	TRANSITIONAL		FRESH			
			ALL	Jinkas, White Dam, Olympia & Dingo	Jackson	Jinkas	Jackson	Dingo	Jinkas South, White dam, & Olympia
1	Base Case Mining and Processing Parameters	Unit	Value	Value	Value	Value	Value	Value	Value
	Breakeven Cut Off Grade (Calculated)	g/t	0.355	0.408	0.434	0.407	0.433	0.417	0.410
	Processing Recovery (Recovery = 1-(Tail grade)/(Head Grade), where Tail grade = m x Head grade + b)	m	0.062	0.036	0.036	0.035	0.135	0.072	0.086
		b	0.018	0.065	0.65	0.067	0.035	0.053	0.046
		Max.	96%	95%	95%	95%	95%	95%	95%
	Mining Capacity	Mtpa	35.0						
	Processing Capacity	tph	518	476	476	450	450	424	450
2	Base Case Geotechnical Parameters	Unit	Value	Value	Value	Value	Value	Value	Value
		Refer to Table 1-4							
3	Mining Economic Parameters	Unit	Value	Value	Value	Value	Value	Value	Value
	Load & Haul	\$/bcm	$\$8.51 + \$0.185 * RL - \$0.00008 * RL$						
	Drill & Blast	\$/bcm	\$2.37	\$3.59		\$4.81			
	Mining Overheads	\$/bcm	\$1.99						
4	Ore Costs (to be included in Pcost)	Unit	Value	Value	Value	Value	Value	Value	Value
	Ore Mining Premium average	\$/t ore	\$0.05						
	Grade Control Cost	\$/t ore	\$0.40						
	ROM Rehandle cost	\$/t ore	\$1.50						
	Miscellaneous	\$/t ore	\$0.60						
	Total OMP Cost	\$/t ore	\$2.55						
4	Processing Economic Parameters	Unit	Value	Value	Value	Value	Value	Value	Value
	Process Fixed costs	\$/hour	\$6,133.95						
	Site G&A	\$/M/year	\$11.1						
	Owner Mining G&A	\$/M/year	\$6.5						
	Processing variable Cost	\$/t ore	\$13.39	\$17.65	\$15.33	\$14.33	\$15.62	\$14.33	\$14.33
	Sustaining Capital (TSF, Plant & Infrastructure)	\$/t year	\$4.0						
	<b>Total Processing Cost</b>	<b>\$/t ore</b>	<b>\$27.79</b>	<b>\$30.77</b>	<b>\$33.09</b>	<b>\$30.51</b>	<b>\$31.81</b>	<b>\$31.34</b>	<b>\$30.86</b>

Pit slope assumptions are summarised in Table 1-4 with the recommendations by 3<sup>rd</sup> Rock Geotechnical Consultants.

**Table 1-4: Pit Geotechnical Parameters for Pit Optimisation**

Area	Domain	Berm Height	Berm Width	Batter Angle	Approx Wall Height	Ramp Allowance	Overall Slope Angle
		m	m	°	m	m	°
All	Oxide	10	6.5	55	20	3	40.4
All	Trans	10	6.5	80	40	6	51.3
Jackson & Olympia	NF1 Fresh	20	8.5	80	40	15	52.6
	NF2 Fresh	20	8.5	90	40	18	56.5
White Dam, Jinkas & Jinkas South	CF1 Fresh	20	8.5	85	150	30	56.7
	CF2 Fresh	20	8.5	90	150	46	56.0
Dingo	SF1 Fresh	20	8.5	85	40	15	56.0
	SF2 Fresh	20	8.5	90	40	30	46.1

The optimisation results show that the resource is highly sensitive to changes in price of gold and overall slopes, and less sensitive to changes in dilution, mining and processing costs, as highlighted in Figure 1-1.

**Figure 1-1: Pit Optimisation Sensitivity Runs (showing changes with ±10% variance in inputs)**



As outlined in Section 5.4.4, the strategy for the project was to provide a mine life of at least 9 years. CZ is the largest pit and provides more than 90% of the total material mined, ore and ounces for the project. The CZ optimisation is therefore the focus for increasing the mine life by selecting a larger shell. Hence, Scenario 103 for the Central Zone shell 44 (Figure 1-2) and Scenario 206 for the South Zone shell 43 (Figure 1-3) provided the basis for shell selection using a revenue factor of 1.14 which equate to a gold price of \$3,420/oz. A summary of the shell inventories is presented in Table 1-5.

**Table 1-5 Detailed Results for Selected Shells**

Scenario	Model	Optimisation Shell	Revenue Factor	Material					
				Ore Mined		Waste	Total	Ounces	Rec Ounces
				TOTAL ORE					
				TOTAL Mt	Au g/t	Mt	Mt	koz	koz
SCN 103	CZ	44	1.14	29.4	1.15	195.7	225.0	1,081	974
SCN 206	SZ	43	1.14	2.7	1.02	22.1	24.8	89	79
SCN 303	NZ	21	0.98	0.3	1.52	2.1	2.3	13	12
<b>TOTAL</b>				<b>32.3</b>	<b>1.14</b>	<b>219.8</b>	<b>252.2</b>	<b>1,183</b>	<b>1,065</b>

**Figure 1-2: Scenario 103 Pit Optimisation Results for CZ**

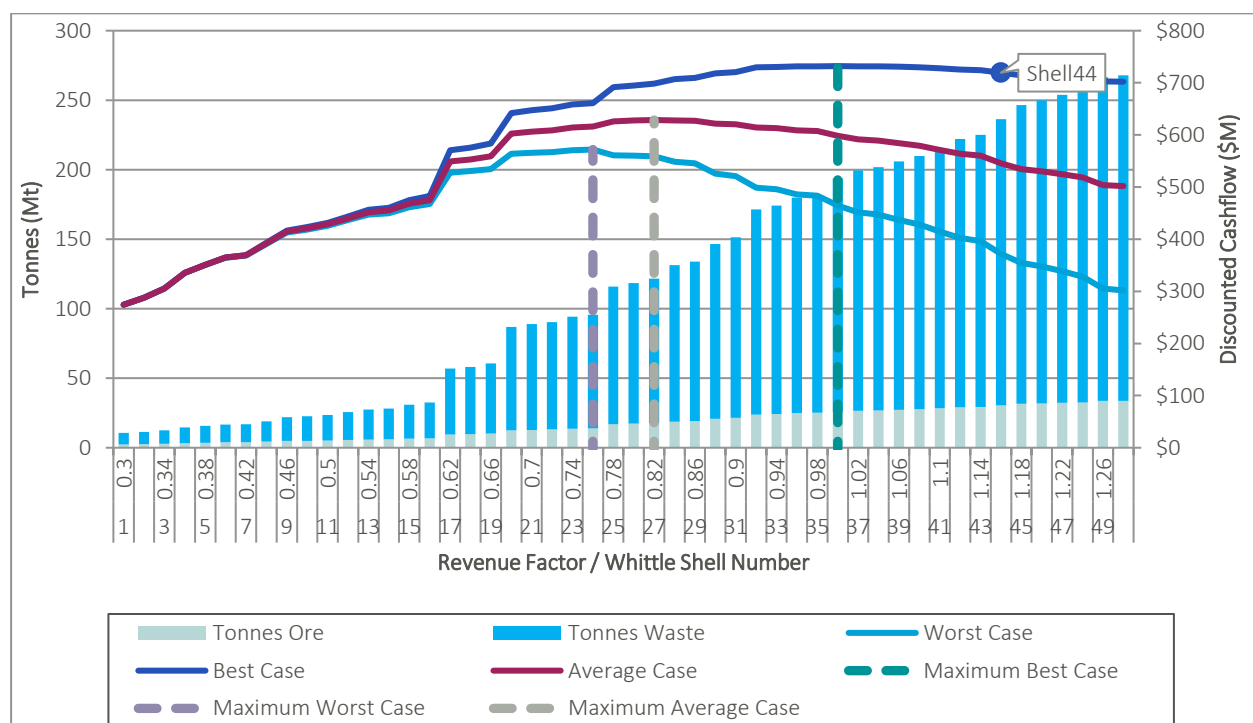
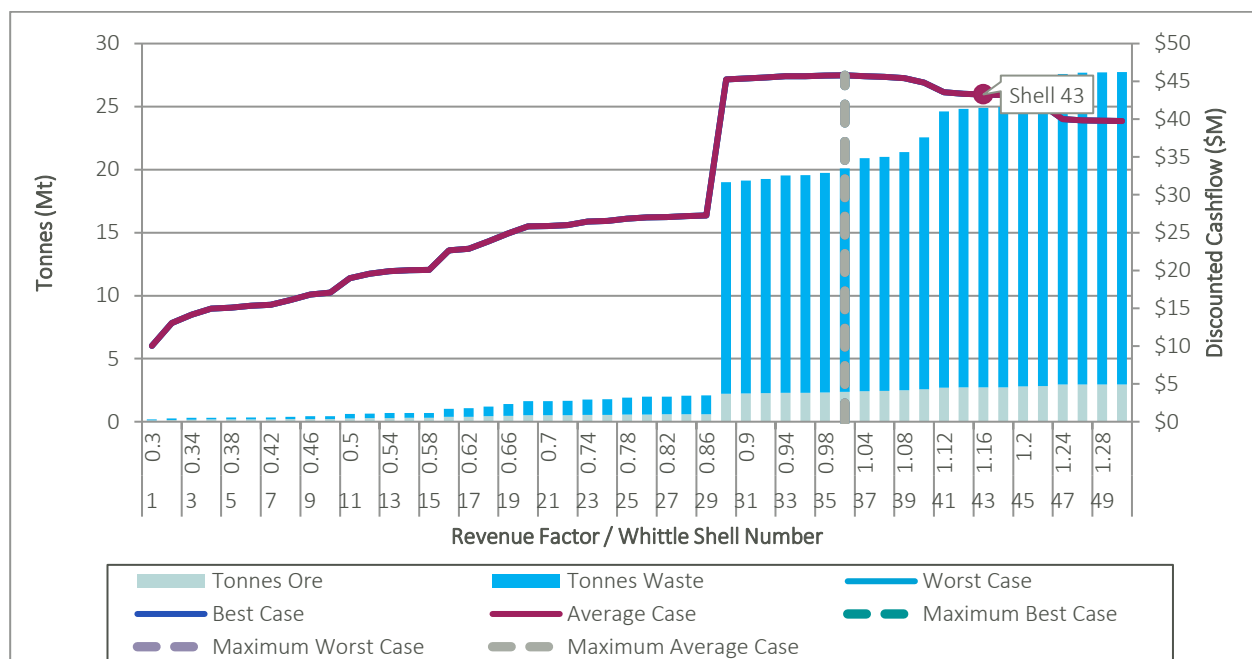


Figure 1-3 Scenario 206 Pit Optimisation Results for SZ



Note: Best, Worst and Average Case were identical due to the short mine life of the pit

## 1.5 Mine Design

Ultimate pit designs were completed using the criteria outlined in Table 1-4 except that 90° batters were revised to 80° and berm widths reduced to retain the inter-ramp slope angle. Ramp designs set at 30 m in width and a 1:10 gradient for dual ramp access, and 16 m wide for single lane access, with the gradient at 1:9. The designs are not for construction and should be reviewed prior to implementation.

The Jinkas pit has an overall length of 1.4 km, maximum width of 450 m and depth 220 m. Jinkas was divided into seven stages with interconnecting ramp systems.

Jackson has an overall length of 2.5 km, maximum width of 320 m and depth of 110 m. It has four interconnected stages, each serviced by a separate ramp.

Olympia is comprised of five small pits up to 150 m wide and 200 m long with depths ranging from 25 m to 40 m.

Dingo is comprised of two separate pits with the larger being 1,100 m by 300 m and 80 m deep, and the smaller being 450 m by 300 m and 50 m deep.

Approvals to access to the NZ pit (Datatine or data1) are in the early stages of negotiations, therefore due to the uncertainty, the pit has not included as part of the DFS.

The inventories for each stage are presented in Table 1-6 including 159 kt at 0.99 g/t of Inferred.

Table 1-6: Inventories by Stage

Phase	Total Ore			Waste	Total Rock	SR
	Tonnes	DAU	Ounces	Tonnes	Tonnes	
jack1	1,821,894	1.19	69,420	12,468,925	14,290,819	6.8
jack2	1,562,241	0.90	45,296	8,772,475	10,334,716	5.6
jack3	741,993	1.11	26,507	4,568,958	5,310,951	6.2
jack4	711,043	1.29	29,411	8,461,608	9,172,651	11.9
Jink1	5,592,610	1.33	238,631	30,045,758	35,638,369	5.4
Jink2	2,332,132	1.22	91,504	26,154,902	28,487,034	11.2
Jink3	5,075,044	1.44	234,891	41,719,369	46,794,414	8.2
Jink4	3,619,438	0.83	96,331	26,891,917	30,511,354	7.4
Jink5	9,002,273	0.97	282,156	50,466,730	59,469,003	5.6
Jink6	1,500,295	0.78	37,627	8,040,163	9,540,458	5.4
olym1	110,797	0.70	2,505	328,599	439,396	3.0
olym2	60,315	1.20	2,325	110,605	170,920	1.8
olym3	102,478	0.90	2,955	510,276	612,754	5.0
olym4	195,594	1.00	6,295	628,746	824,340	3.2
olym5	38,370	0.85	1,043	258,632	297,001	6.7
ding1	2,304,158	1.02	75,708	18,525,792	20,829,950	8.0
ding2	566,460	0.86	15,624	4,186,881	4,753,341	7.4
<b>Total</b>	<b>35,337,135</b>	<b>1.11</b>	<b>1,258,229</b>	<b>242,140,336</b>	<b>277,477,471</b>	<b>6.9</b>

The west walls of the pits follow the footwall of the ore body which required an adjustment to the berm width at each 10 m bench increment. When possible, the ramps were placed on the footwall to reduce mining waste on the highwall. This was especially important for the Jinkas pit as the pit shell was constrained on the eastern side by the lease boundary. Using this method, the final bench of the Jinkas pit design was below the base of the pit shell, thereby adding additional ore to the inventory without compromising the final wall constraint. Other pits were designed with ramp systems aimed at minimising haul distances to the ROM and waste dumps.

The Dingo 1 pit was constrained not by the tenement boundary but by the Rifle range exclusion zone. In order to maximise recovery of the resource from this pit, the east side of the pit will be backfilled with waste from Dingo 2 to move the zone of potential failure, and abandonment bund, outside the exclusion zone.

The reconciliation between design and selected pit shells is presented in Table 1-7, which shows that the final designs have captured approximately ~8% more ore at the cost of ~12% more waste resulting in a small increase in strip ratio by ~4%. The additional ore has an average diluted gold grade (DAU) of 0.95 g/t and adds 80 koz of contained gold at a cost of ~\$2,200/oz operating cost.

Table 1-7 Design to Pit Shell Reconciliation

Design vs Shell	Material	Grand Total				
		Volume (Mbcm)	Mass (Mt)	DAU (g/t)	Au (koz)	Strip Ratio
Total design	Total Ore	12.8	35.3	1.11	1,258	
	Waste	96.3	242.1			6.85
	Total Rock	109.1	277.5			
Total optimisation shell	Total Ore	11.9	32.8	1.12	1,180	
	Waste	86.1	216.9			6.62
	Total Rock	98.0	249.7			
Difference	Total Ore	7.7%	7.8%	-1.1%	6.6%	
	Waste	11.9%	11.6%			3.6%

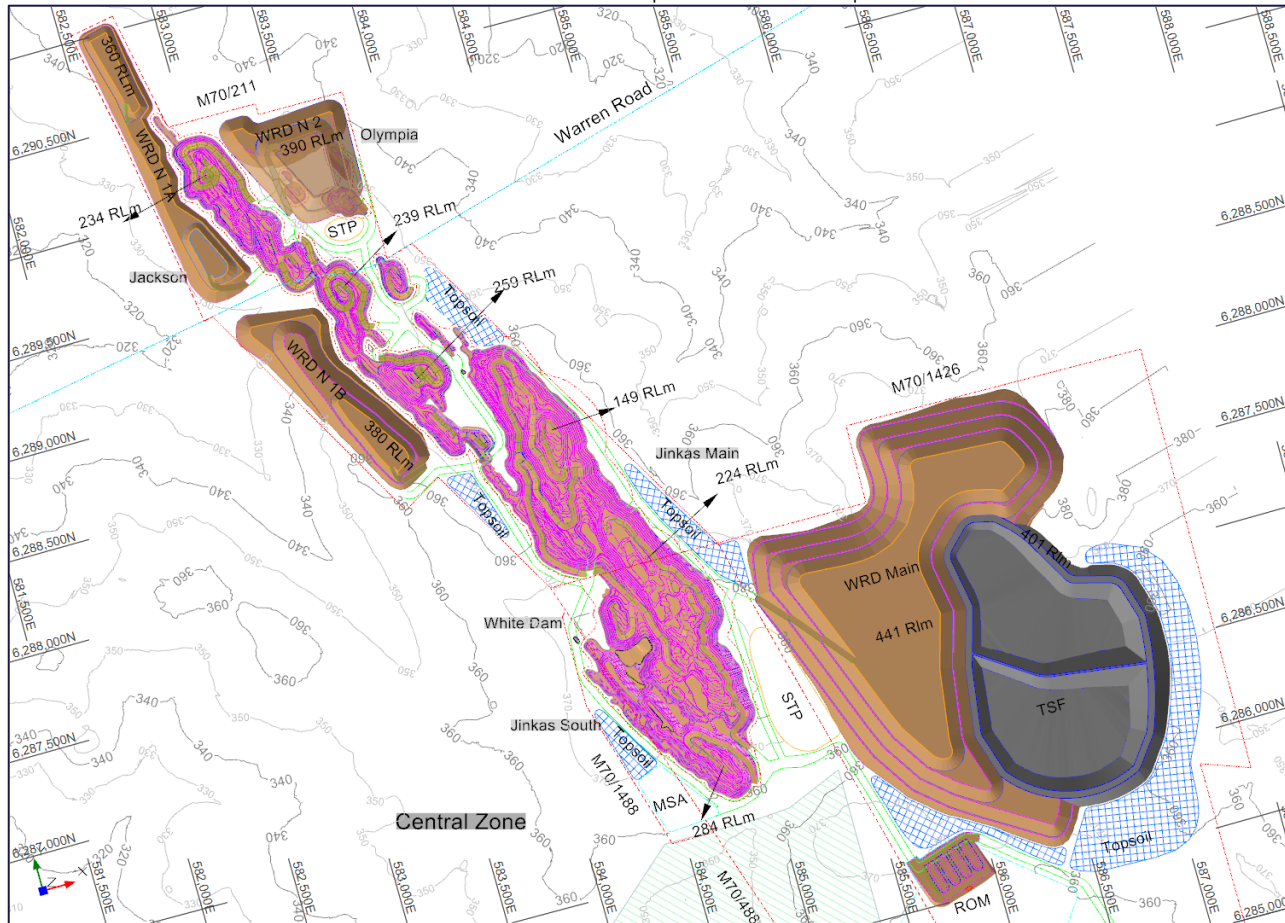
A high level overview of the stages are shown in Figure 1-4. The waste rock dumps have been designed with 20.0 m high lifts using 18° batter and 8 m wide berms to give an overall slope of 16°. The Main waste dump has a maximum dump height of 80 m above ground level and is located adjacent to the ROM pad and stockpile as presented in Figure 1-5.

Figure 1-4: Design Stages



Figure 1-5: Waste Rock Dump Design and Overall Layout

Central Zone Ultimate pits and waste dumps



Dingo Ultimate pits and Waste dumps



## 1.6 Mine Production Schedule

The Katanning Project pits were scheduled using Hexagon MinePlan Schedule Optimizer™ (MPSO), an advanced mine scheduling tool that optimizes cut-off grades, phase sequencing, and haulage while considering operational constraints. MPSO integrates multiple objectives to deliver practical and efficient schedules that maximize project value.

The schedule assumes that the mining contractor will mobilize during the two quarters before plant production begins, with initial stripping, clearing, and pre-strip-mining taking place in the four months leading up to plant production.

To facilitate scheduling, the ultimate pit was split into eighteen stages to ensure ore continuity at reasonable strip ratios. It should be noted Jinkas Stage 1 is split into a west (Stage1A) and east (Stage1B) to aid in balancing mining and maintaining ore supply over the initial monthly periods.

Mine scheduling requires balancing conflicting objectives and constraints. For the Katanning Project, the primary goal is to maximize Net Present Value (NPV) while adhering to practical mining limitations. The key considerations include:

- Conducting pre-production mining to produce approximately 5 Mt of total transitional and fresh material for tailings dam construction. The oxide waste will be used to construct the ROM pad.
- Prioritizing high-grade material extraction in the early years.
- Maintaining a production rate averaging at least 140 koz/year for the first four years to maximise cash flow during the payback period.
- Deferring mining in the Jackson area until the end of Year Three to delay the capital cost for diversion of the Warren Road.
- Limiting oxide content in the process plant feed to a maximum of 25% per period, as laboratory results indicate processing issues when exceeding this threshold.
- Restricting Jackson Stage 1 ore to no more than 8% per period, due to its higher processing cost, lower recovery rate, and increased risk associated with reactive pyrrhotite.
- Enforcing a maximum bench turnover rate per stage at a maximum of 80 m per year.

The stockpiling strategy was primarily based on grade streaming to maximise cashflow with a secondary objective to control the level of oxide material in the ore feed to avoid “flooding” of some processing circuits. A third objective was to limit the amount of reactive transitional material found in Jackson that caused excessive consumption of reagents. To achieve this, the model was developed with a two ore bins (HG > 0.8 g/t and LG > 0.4 g/t) and four ore types (Oxide, Trans, React Trans & Fresh).

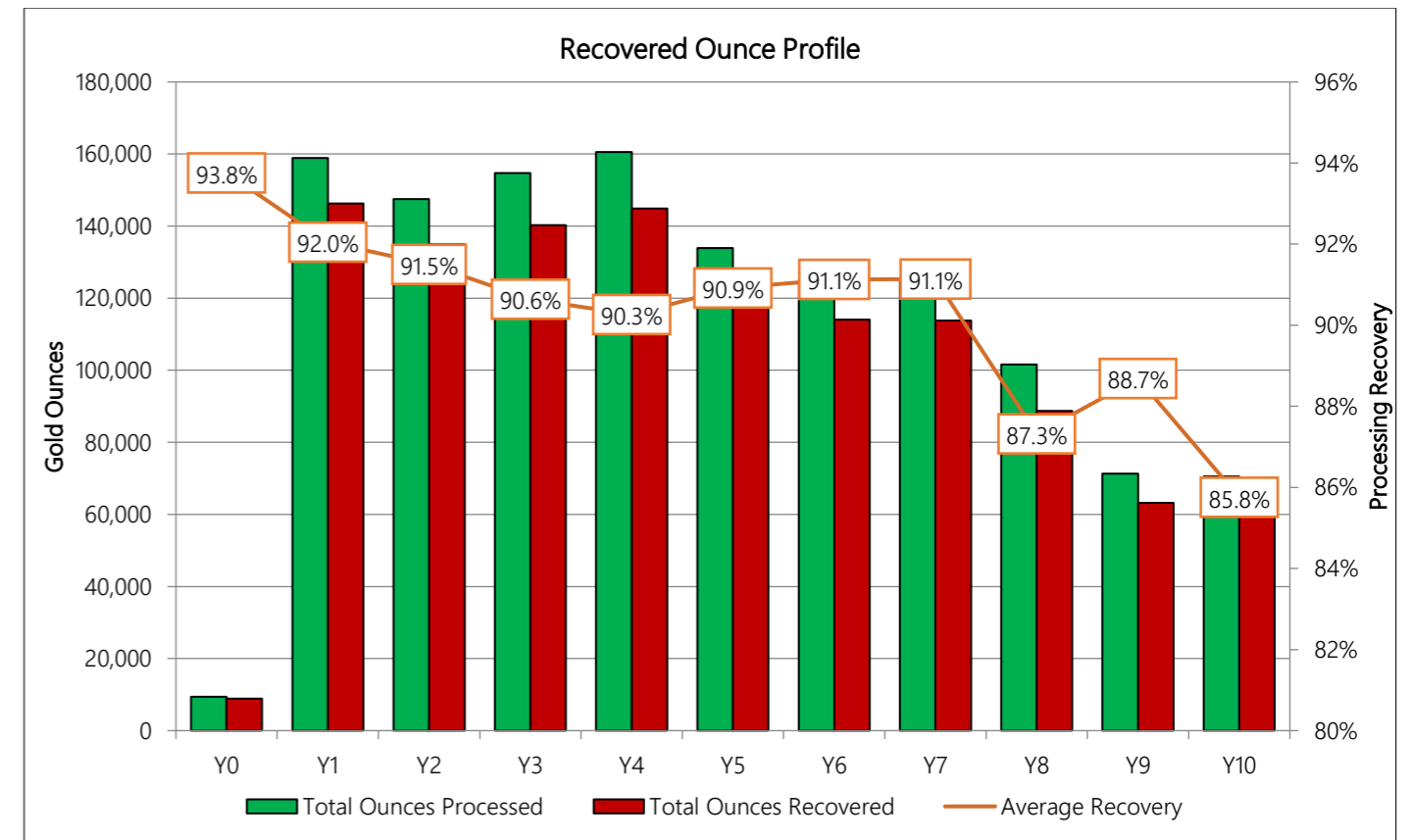
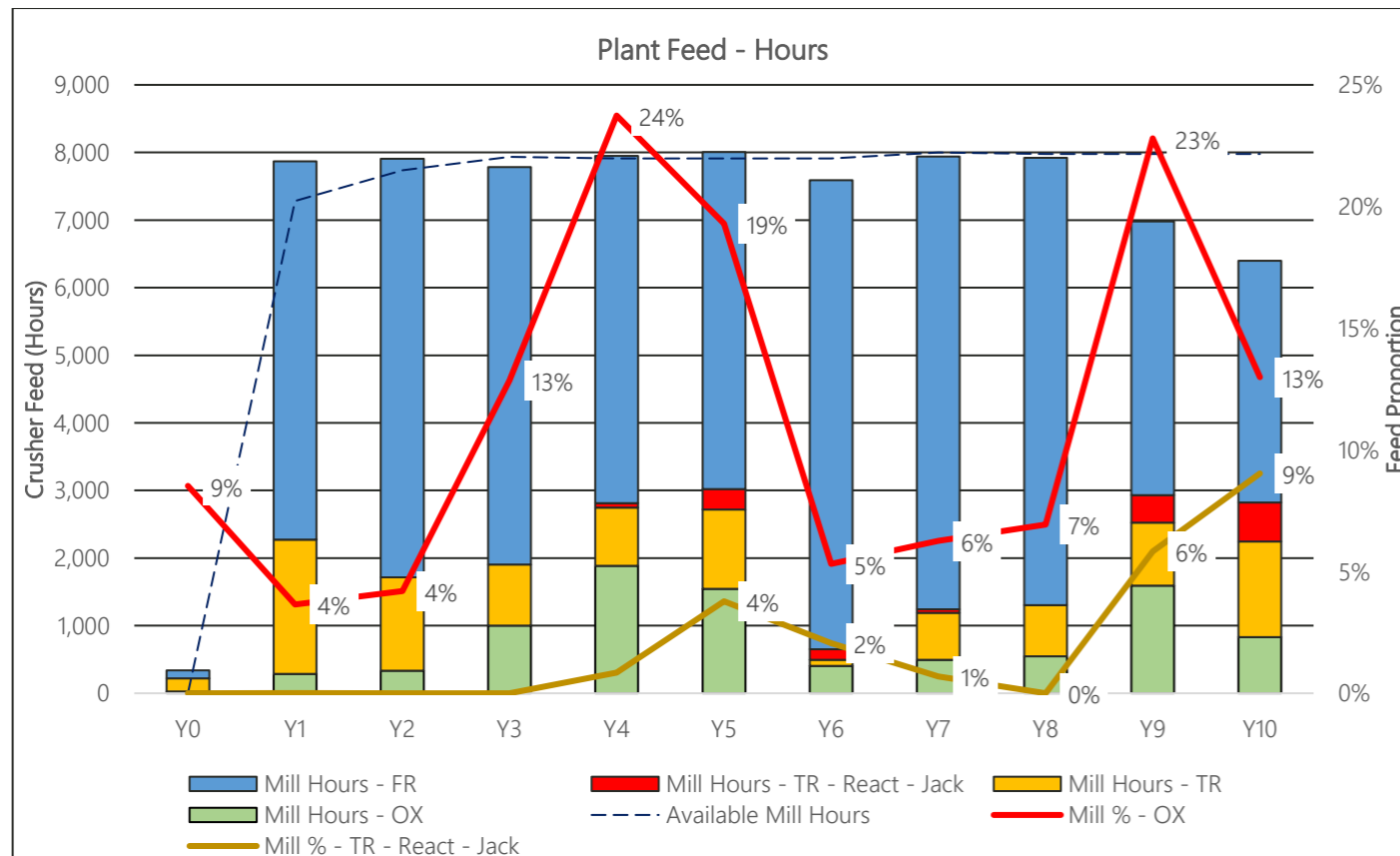
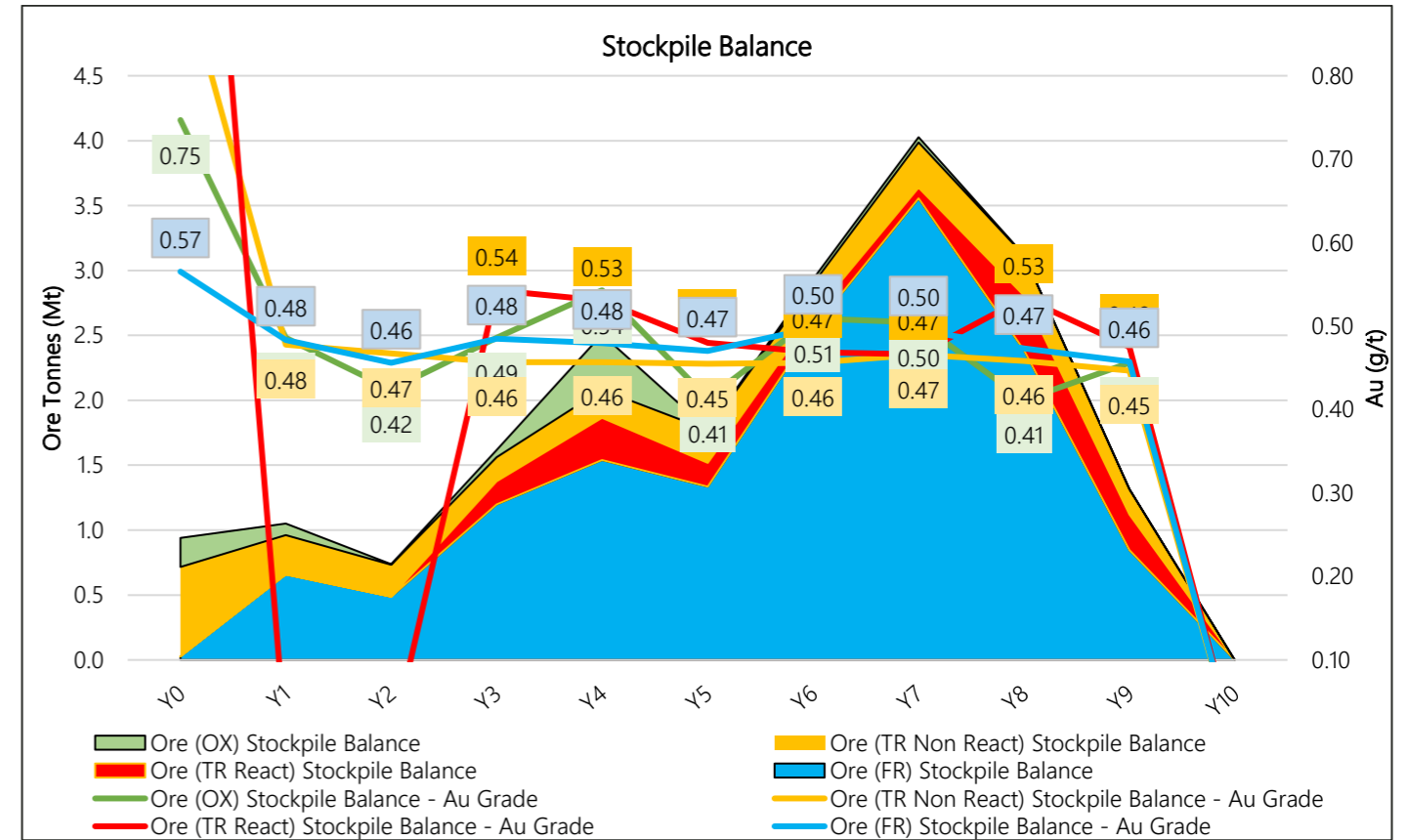
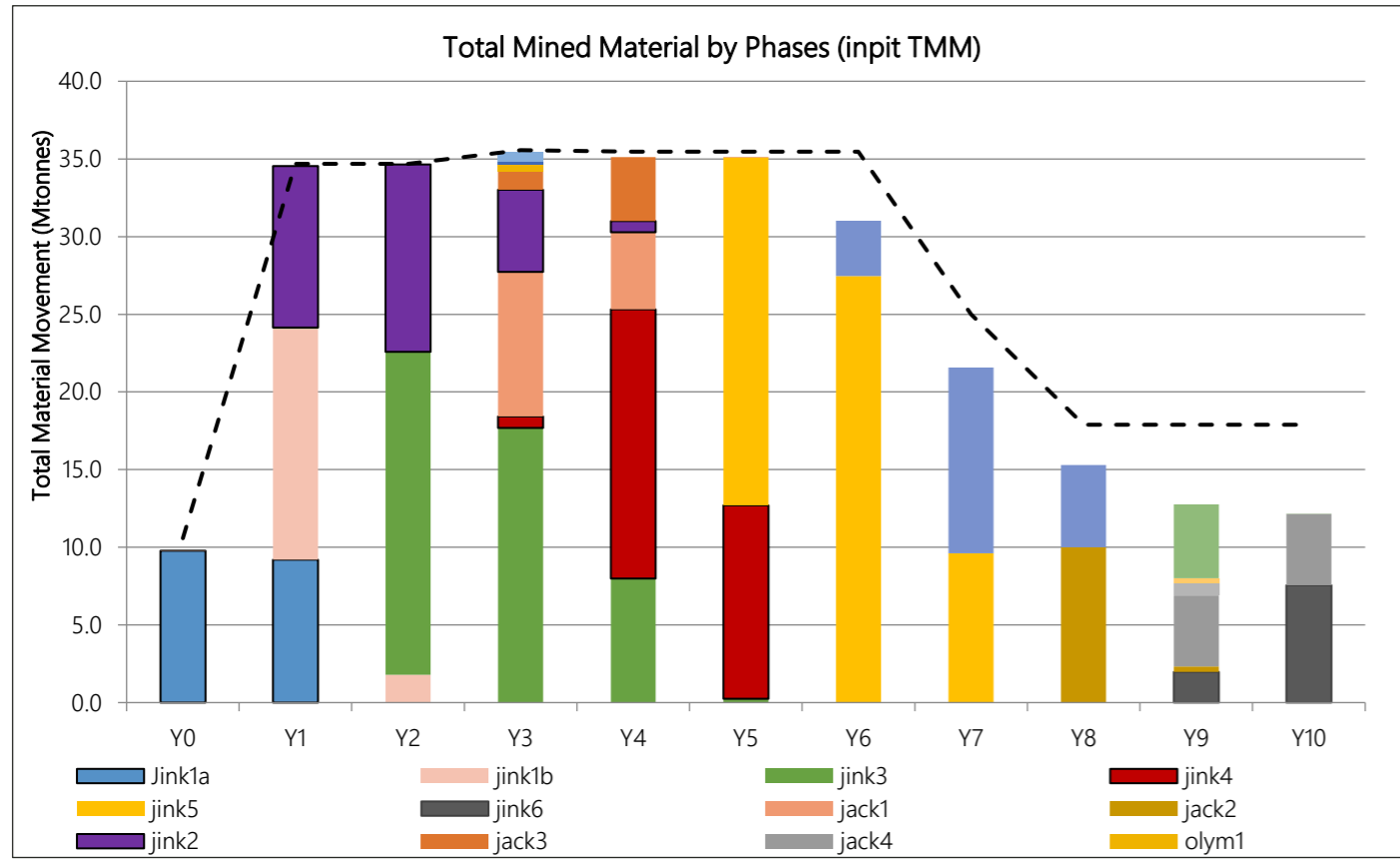
Material movement rates were based on excavator productivity assuming 85% availability and 85% utilisation for 345 days assuming 18 days lost to wet weather and 2 days to public holidays. Productivity for the 250 t excavator was 12 Mtpa in oxide and 13.2 Mtpa in trans/fresh rock. The 120 t ore digger productivity was based on 4.9 Mtpa in oxide and 5.0 Mtpa in trans/fresh rock.

Table 1-8 and Figure 1-6 summarise the schedule results.

Table 1-8: KGP Annual Mining Schedule

	Items	Units	Total	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
TOTAL Mined	Rock mined Ex-pit	Mt	277.5	9.8	34.5	34.7	35.4	35.1	35.1	31.0	21.6	15.3	12.8	12.2
		Mbcm	109.1	4.4	12.7	12.7	13.4	15.4	13.9	11.4	8.1	5.9	6.2	5.1
	Waste Mined	Mt	242.1	8.7	30.8	31.3	31.0	30.5	32.1	26.4	16.9	12.7	11.3	10.5
		Mbcm	96.3	3.9	11.4	11.6	11.8	13.6	12.8	9.9	6.5	4.9	5.5	4.5
	Strip Ratio	W:O	6.9	7.9	8.3	9.5	6.9	6.6	10.7	5.7	3.6	4.8	7.7	6.3
	Ore Mined	Mt	35.3	1.10	3.72	3.30	4.47	4.62	3.01	4.61	4.71	2.64	1.47	1.67
		Mbcm	12.8	0.4	1.2	1.1	1.6	1.8	1.1	1.6	1.6	0.9	0.7	0.7
Ore Mined - Au Grade	g/t	1.11	1.02	1.24	1.34	1.18	1.18	1.25	0.98	0.94	1.01	0.90	0.95	
Direct Feed	Ore	Mt	26.0		1.8	2.7	3.3	3.4	2.7	3.1	3.6	2.3	1.4	1.7
	Au Grade	g/t	1.28		1.8	1.5	1.4	1.4	1.4	1.2	1.1	1.1	0.9	1.0
	Contained Ounces	koz	1,072		103.7	130.9	150.6	155.4	115.3	119.1	124.2	79.9	41.9	51.1
	Recovery	%	90.8%		92.6%	92.0%	90.8%	90.4%	91.4%	91.4%	91.2%	87.3%	90.5%	86.8%
Stockpiled	Ore	Mt	9.35	1.1	1.9	0.6	1.1	1.2	0.4	1.5	1.1	0.3	0.0	
	Au Grade	g/t	0.62	1.02	0.74	0.56	0.51	0.52	0.47	0.52	0.49	0.55	0.48	
Reclaimed	Ore	Mt	9.35	0.2	1.8	0.9	0.3	0.3	1.1	0.4	0.0	1.2	1.9	1.3
	Au Grade	g/t	0.619	1.85	0.97	0.55	0.49	0.54	0.53	0.50	0.53	0.54	0.49	0.46
	Contained Ounces	koz	186	9.3	55.2	16.6	4.1	5.1	18.6	6.1	0.6	21.7	29.4	19.4
	Recovery	%	88.0%	93.8%	91.0%	87.6%	85.7%	85.4%	87.9%	85.1%	85.5%	87.4%	86.1%	83.0%
Processed	Ore	Mt	35.3	0.2	3.6	3.6	3.6	3.7	3.7	3.4	3.6	3.5	3.3	3.0
	Au Grade	g/t	1.11	1.83	1.37	1.27	1.34	1.34	1.11	1.13	1.07	0.89	0.68	0.74
	Total Ounces Processed	koz	1,258	9.4	158.9	147.5	154.7	160.5	133.9	125.2	124.8	101.6	71.3	70.6
	Total Ounces Recovered	koz	1,137	8.9	146.3	134.9	140.3	144.9	121.7	114.1	113.8	88.7	63.2	60.5
	Average Recovery	%	90.4	93.8	92.0	91.5	90.6	90.3	90.9	91.1	91.1	87.3	88.7	85.8
	Inferred proportion	%	0.45				0.05	0.06	0.69	0.40	0.18			2.13

Figure 1-6: Annualised Schedule Results



## 1.7 Mine Cost Estimation

The mining cost estimate for the Project assumes that the Ausgold engage a suitably qualified contractor for the mining services including mine production, maintenance and supervisory functions, with technical services and management functions performed by Ausgold.

The costs are based on the physical quantities derived from each of the mine plans described in Section 8 and then developed from a combination of the physical quantities together with mining contract rates. The primary sources were:

- Mining Contract submissions from Mar 2025.
- A fuel price of \$1.10 per litre excluding GST and Diesel fuel rebates.

Total capital and operating costs were estimated at \$1,471 M or \$5.30/t mined.

Capital costs totalled \$43.3 M comprised of:

- Owner Equipment - \$0.61 M
- Contractor Site Establishment - \$2.90 M
- Contractor Mobilisation - \$2.87 M
- Pre-production mining cost - \$36.89 M

Total operating cost for the life of mine were estimated at \$1,428 M as summarised in Table 10-19.

Table 1-9 Annual Mining Cost Summary by Cost Centre

OPEX (\$M)	Y0 (\$M)	Y1 (\$M)	Y2 (\$M)	Y3 (\$M)	Y4 (\$M)	Y5 (\$M)	Y6 (\$M)	Y7 (\$M)	Y8 (\$M)	Y9 (\$M)	Y10 (\$M)	Total (\$M)
Clear & Grub	0.1	0.4	0.3	0.3	0.4	0.1	0.1	0.1	0.0	0.09		1.8
Topsoil removal	0.4	2.8	2.1	2.0	3.1	0.4	0.8	0.9	0.4	0.6		13.4
Mine Haul Roads	0.3			0.7	0.1	0.1		0.7				1.9
Owner Overheads	0.4	4.7	4.7	4.7	4.7	4.7	4.7	4.5	4.5	3.8	3.8	45.1
Contractor Oheads	1.2	14.0	14.0	14.0	14.0	14.0	14.0	10.5	10.5	7.0	7.0	120.4
L&H Waste	7.6	81.8	76.4	76.5	76.4	86.5	80.3	42.5	28.5	28.5	30.6	615.7
L&H Ore	1.8	9.9	9.5	13.6	12.9	7.1	12.8	14.3	8.0	4.7	4.6	99.2
D&B Waste	1.4	27.3	29.2	30.5	25.7	27.9	29.0	15.4	10.2	8.5	7.7	212.8
D&B Ore	0.3	3.2	2.8	3.7	3.5	2.4	4.2	4.0	2.0	0.9	1.2	28.2
Fuel Consumption	1.9	26.6	25.3	26.1	24.2	26.2	30.2	15.3	12.9	8.3	7.1	204.1
Ore Rehandle	0.4	5.6	4.4	3.8	4.0	5.3	3.8	3.1	4.7	6.6	2.5	44.2
Grade Control	0.6	3.1	2.8	3.7	5.0	2.8	4.6	4.7	2.3	2.7	2.0	34.3
Pit Dewatering	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	1.8
Mine Rehabilitation						0.5				0.3	2.0	2.8
Demobilisation			0.03		0.03		0.02	0.3	0.1	0.2	1.5	2.2
<b>TOTAL (\$M)</b>	<b>16</b>	<b>180</b>	<b>172</b>	<b>180</b>	<b>174</b>	<b>178</b>	<b>185</b>	<b>117</b>	<b>84</b>	<b>72</b>	<b>70</b>	<b>1,428</b>

## 1.8 Financials

Table 1-10 summarises the EBITDA financials related to the open pit by year. Total operating costs inclusive of mining, processing, and selling costs are A\$2,507 M and together with the total capital costs of \$368 M, the total cost base is \$2,875 M. Total revenue is \$3,412 M based on 1,137 koz of gold recovered using the base gold price of \$3,000/oz. Hence, a final cashflow before taxes and depreciation of \$537 M can be generated with a project IRR of 33% and a payback within just over 4 years.

All In Sustaining Costs (ASIC) are \$2,205 per recovered ounce for the project. Figure 1-7 shows the cost per ounce over the life of the project and Figure 1-8 outlines the cashflow and cumulative discounted cashflow for the project.

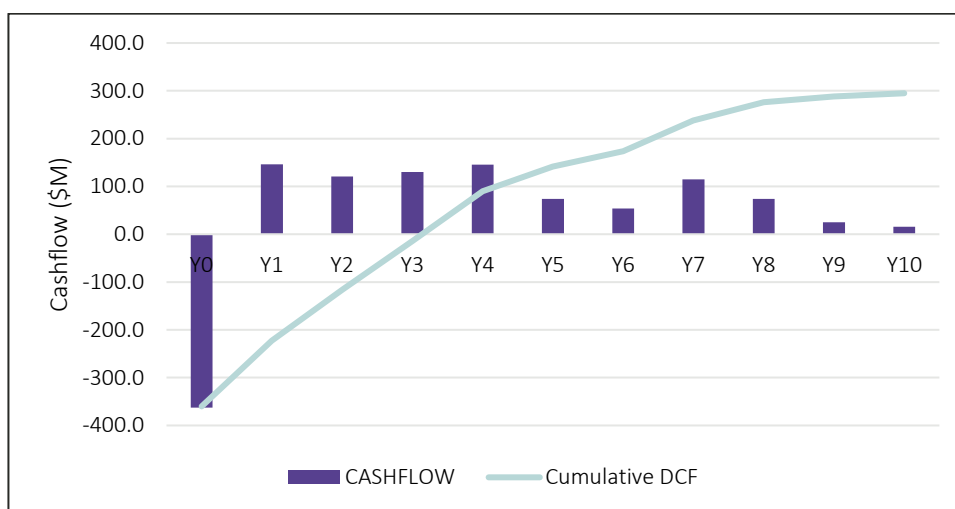
Table 1-10: Open Pit Project Financials

Cost Centre	Units	Amount
Revenue	\$M	3,411.6
<b>Operating Costs</b>		
Mining Costs	\$M	1,427.9
Processing Costs	\$M	985.1
Selling Costs	\$M	93.6
Total OPEX	\$M	2,506.6
<b>Capital Costs</b>		
Mining	\$M	43.3
Processing	\$M	325.0
Total CAPEX	\$M	368.3
Cashflow	\$M	536.7
Discounted Cashflow 5%	\$M	294.8
<b>IRR</b>	%	33%
<b>Average cost per pound (ASIC)</b>	\$/oz	2,205
Payback period	Years	4
Process Mine Life	Years	10

Figure 1-7: Annual AISC per ounce



Figure 1-8: Cumulative Discounted Cashflow



## 1.9 Risks and Opportunities

### 1.9.1 Execution Risks

The following risks need to be considered as part of the next phase of work and include:

- The 90° batter slopes recommended by 3rd Rock Consulting (Section 1.4) are impractical for design purposes, therefore, for the DFS pit designs, the 90° batter slopes were adjusted to 80° with a subsequent reduction in the berm width to 5.0 m to maintain inter-ramp slope angle. In practice, this berm is too narrow to effectively catch and contain small scale failures, and a steeper batter slope and wider berm would need to be considered in operations. .
- The Warren Road crossing requires detailed design and development of a traffic management plan to minimise the risk to the public.

- The mine is surrounded by farmland which will require management of dust and sound particularly at night.

### 1.9.2 Mining Cost Risks

As with all mining projects, there is a risk that mining costs will be more than anticipated in this cost estimate due to the following:

- The rock is strong and highly competent which may not fracture as well as modelling has shown using modified Kuzram methods. Improved fragmentation may require tighter drilling patterns or higher density explosives.
- Inflationary influences causing the cost of fuel, parts and labour to increase.

### 1.9.3 Execution Opportunities

There are numerous opportunities identified with the development of the project that will be evaluated and integrated with existing or continued technical work including:

- Feedback from one of the Mining Contractors recommended changing the waste drilling hole diameter from 172 mm to 165 mm as this increases the options for drill selection and provides consistence in blast design.
- Two of the contractors recommended an increase in truck size from 140 t to 180 t. The potential cost savings from the larger trucks is likely to provide a small mining cost reduction.
- Due to space limitations, the Main waste dump is located to the south east of the Jinkas pits. Expanding the tenements to the East of the Jinkas/White Dam will reduce the haulage distance for waste.
- Backfilling of the Olympia pits has been exploited to minimise ex-pit hauls and mining costs from Jackson, however, there is potential for a larger pit at Olympia at prices above \$4000/oz. Expansion of the tenements to provide additional areas for waste storage is recommended to avoid potential sterilisation of this ore.

## 2 INTRODUCTION

### 2.1 Background

Ausgold Limited (AUC) is a Perth-based gold exploration and development company. Its primary focus is the 100%-owned Katanning Gold Project (KGP), which covers >4,000 km<sup>2</sup> of the Katanning greenstone belt in south-western Western Australia. Gold mineralisation at the KGP has been identified over 20 kilometres of strike and the project contains a JORC 2012-compliant Mineral Resource of 89 million tonnes at 1.06 g/t gold for 3,040,000 ounces.

AUC is planning to develop a 3.6 Mtpa processing facility at site. Ore will be processed using single stage crushing, grinding and conventional gold processing Carbon in Leach (CIL) technology to produce gold bullion which will be sold to a nearby gold refinery for further processing. The process plant will also produce bullion via gravity gold concentration, intensive cyanidation and electrowinning.

### 2.2 Location

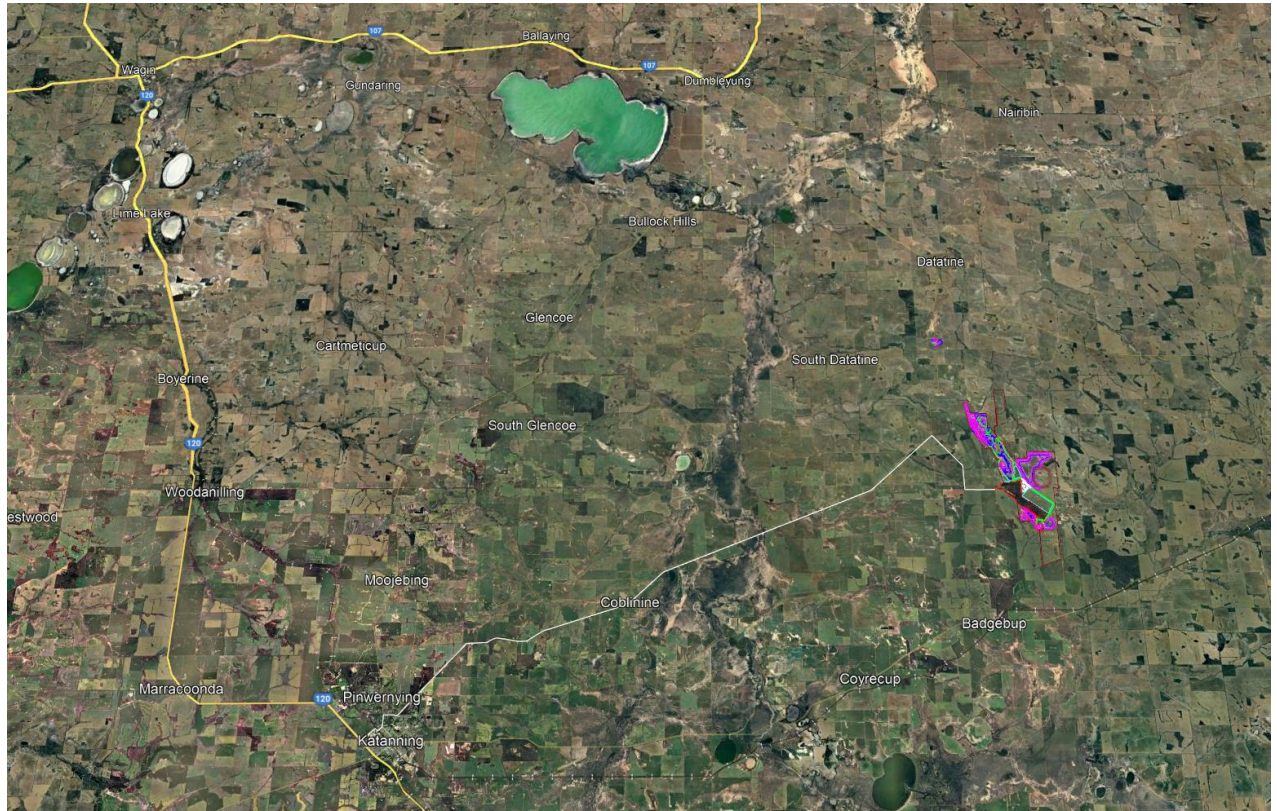
The Katanning Gold Project is located approximately 275 km south-east Perth in Western Australia, as shown in Figure 2-1.

Figure 2-1: Katanning Gold Project Location



Access to site from Katanning is via the 40 km along the sealed Warren road to the Wolyaming Road turnoff and then 4 km by a final unsealed section to access the mine site. The Ausgold project area is shown below in Figure 2-2.

Figure 2-2 Access to the KGP Site



## 2.3 Geology

Regionally, the KGP is located within the Archean aged, metamorphosed greenstone belt which is located in the Southwest Terrane of the Yilgarn Craton. Included in the terrane are granites, greenstones and schists, some of which have been subjected to high-grade metamorphism and structural deformation.

The KGP mineralisation is hosted within the greenstones and the geochemistry of these volcanic rocks suggests they erupted into an Andean style magmatic arc setting. The volcanic rocks are considered calc-alkaline basaltic to rhyolitic lavas, which were later metamorphosed to create mafic through to felsic, granulite gneisses. The granulite rocks were later intruded locally by sheet-like quartz monzonite.

The below major rock types are encountered at the KGP:

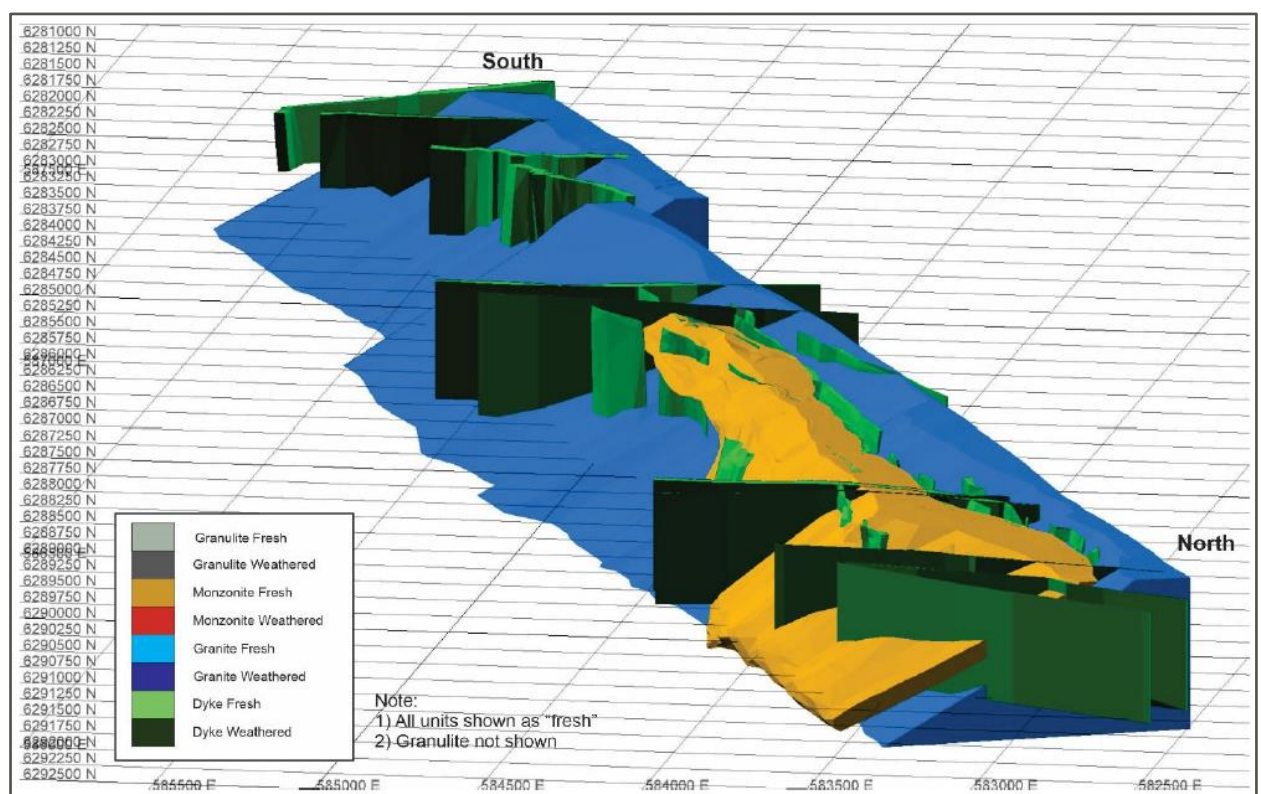
- Granulites (various types).
- Granite.

- Quartz Monzonite.
- Dolerite Dykes.

Footwall granite is encountered primarily at Dingo. These, along with dykes and quartz monzonite have been modelled by Ausgold for the Central and Southern Zones.

Figure 2-3 below shows an isometric view of local geology looking northeast to southwest (source: 3RC Final DFS Geotechnical Report).

Figure 2-3 Isometric view of local geology looking from northeast to southwest



(source: 3RC Final DFS Geotechnical Report)

The footwall granite is a coarse grained, light to slightly pinkish coloured igneous rock composed mainly of feldspars and quartz. The unit is unmineralised and only fresh granite was intersected in drillholes with all granite being classified as very strong rock.

Most of the rocks in which the KGP pits will be mined are Granulites, which are a class of high-grade metamorphic rocks of the granulite facies that have experienced high-temperature and moderate-pressure metamorphism. The following four different types of Granulites were identified:

- Mafic Granulites
- Felsic Granulites
- Intermediate Granulites
- Speckled Granulite.

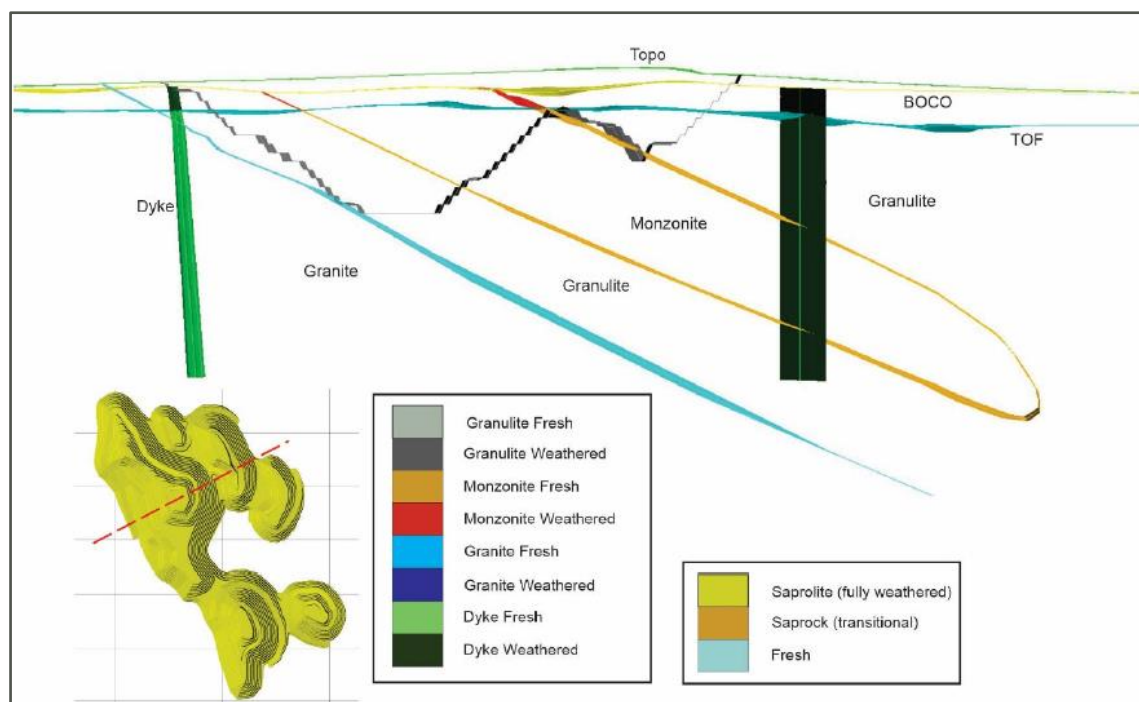
The Granulites host all of the mineralisation. Although relic bedding is occasionally visible in the core, it does not seem to create planes of weakness that cause core breakage. Apart from weathering and mineralisation, there are no noticeable differences in the geotechnical properties across the various types of Granulites, thus they are treated as a single lithological unit.

Monzonite is an intermediate plutonic igneous rock composed of nearly equal amounts of alkali feldspar and sodic plagioclase, along with minor quartz or feldspathoids. It has intruded into the Northern Zone of the deposit, potentially affecting the Jackson, White Dam, and Jinkas Main pits. The unit is unmineralized, with predominantly fresh monzonite encountered in drill holes. All fresh rock was classified as "very strong rock."

KGP hosts a number of extensive dolerite dykes, which are typically fine to medium-grained intrusive rocks that form sub-vertical structures cutting across all surrounding lithologies. These dykes tend to be narrow, usually not exceeding a few metres in width, resulting in relatively few core intercepts. The contacts and margins of the dykes often exhibit a finer texture, and more jointing compared to the coarser-grained interior, which has fewer joints. Notably, the dolerite lacks any associated rock fabric (foliation).

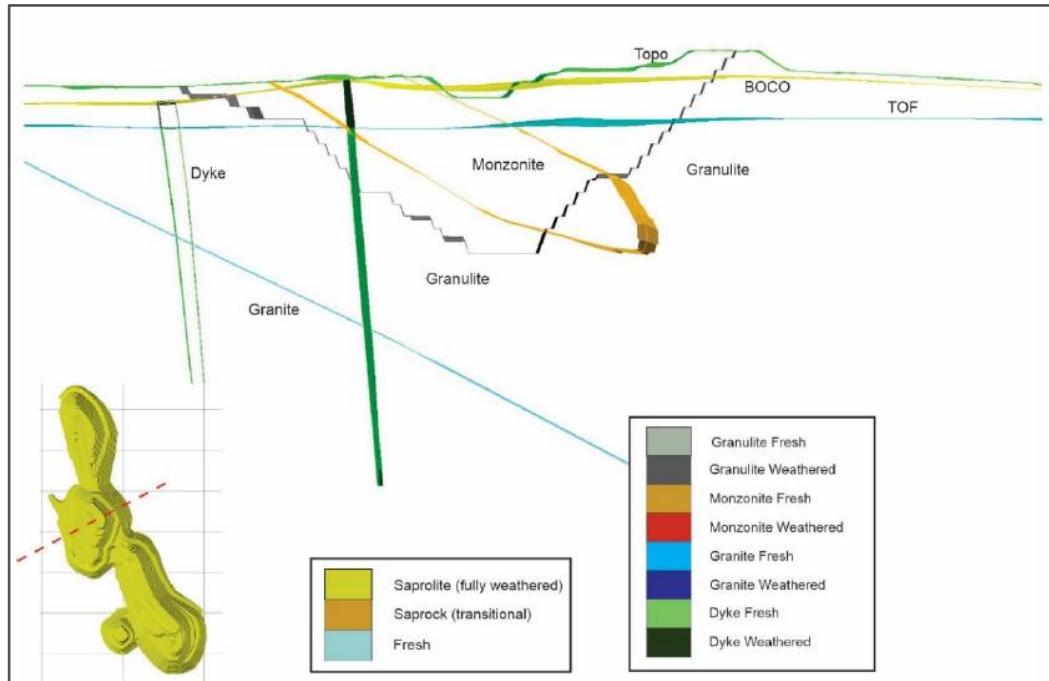
Figure 2-4 and Figure 2-5 below demonstrate the distribution of rock lithologies, including weathering on the pit walls for the Northern Pits – Jackson and Jinkas Main.

Figure 2-4: Cross-section through geology – Jackson Pit



(source: 3RC Final DFS Geotechnical Report – not DFS pit design)

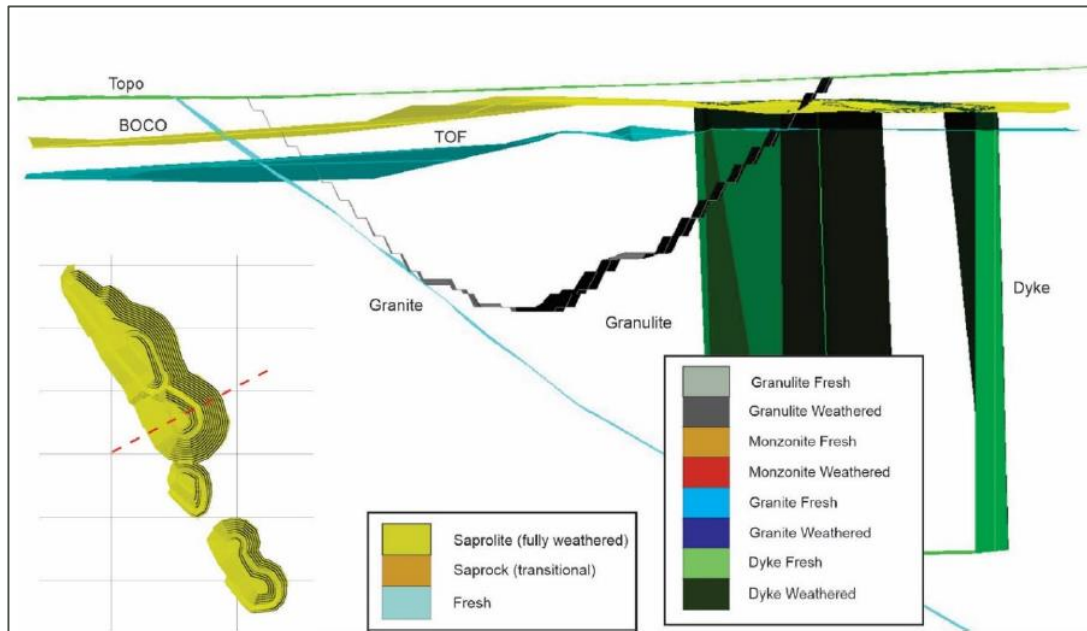
Figure 2-5: Cross-section through geology – Jinkas Main Pit



(source: 3RC Final DFS Geotechnical Report – not DFS pit design)

Figure 2-6 below demonstrates the distribution of rock lithologies, including weathering on the pit walls for the Southern Pit - Dingo.

Figure 2-6: Cross-section through geology – Dingo Pit



(source: 3RC Final DFS Geotechnical Report – not DFS pit designs)

## 2.4 Historical Mining

Glengarry Mining NL conducted small-scale open-pit mining operations between 1995 and 1997. In certain areas, excavation reached depths of around 35 metres. Mining primarily targeted the Oxide material using 'free-dig' methods, while drill and blast techniques were necessary for the more solid Saprock (Transitional) materials at greater depths. By 1997, the mine had produced approximately 20,000 ounces of gold from the Jinkas and Dingo pits. Since then, no further mining has taken place, and the pit walls remain exposed to the present day as shown in Figure 2-7.

Figure 2-7 Western Wall of Jinkas Pit wall showing well developed foliation



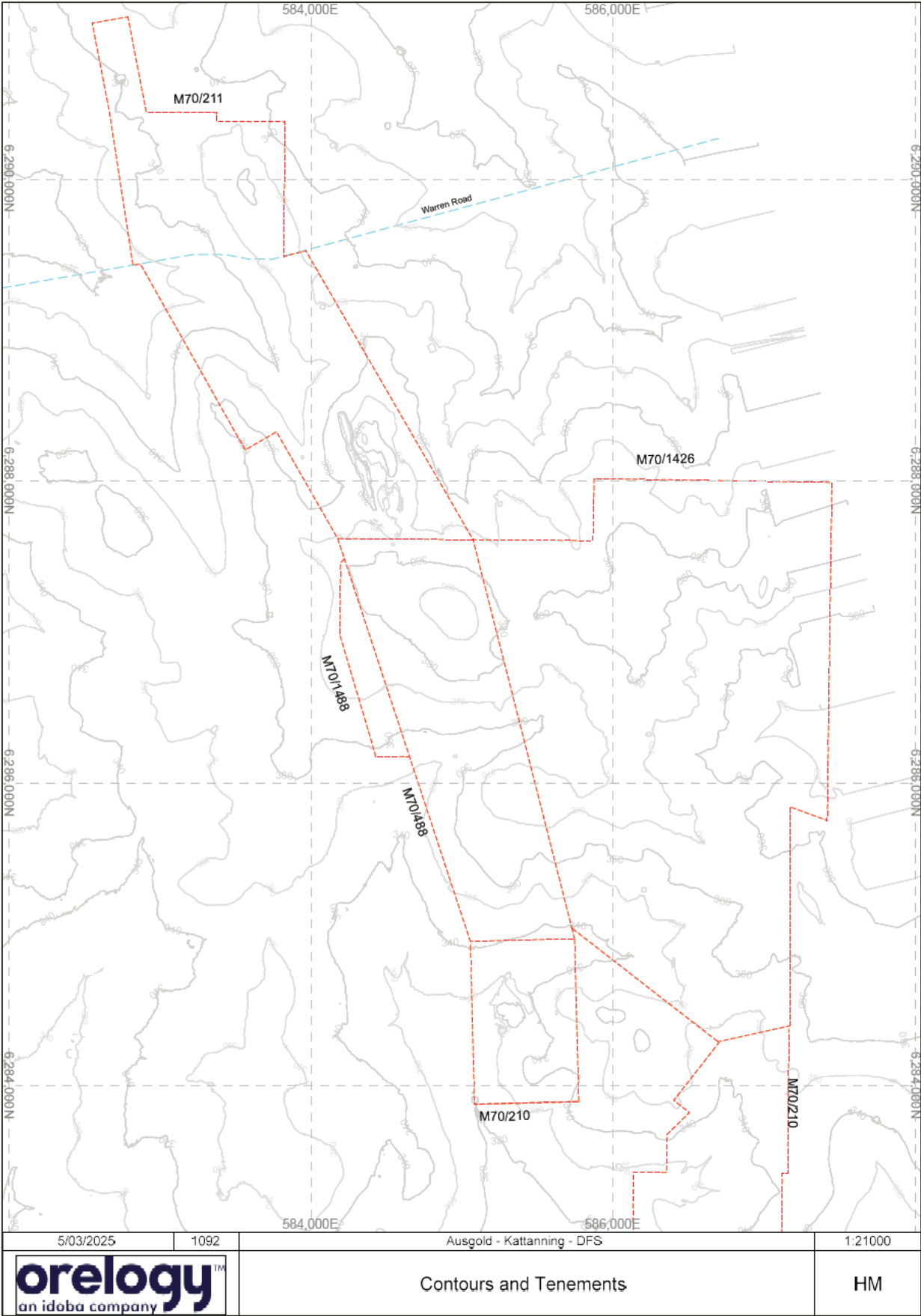
## 2.5 Tenement Boundaries

The KGP is constrained within the existing Ausgold mining leases, M70/210, M70,211, M70/488, M7/1426 & M70/1427 as shown in Figure 2-8. Application for a mining lease will be required for the Dingo pit as part of the pit lies in E70/2928, outside the existing mining lease boundaries.

## 2.6 Topography

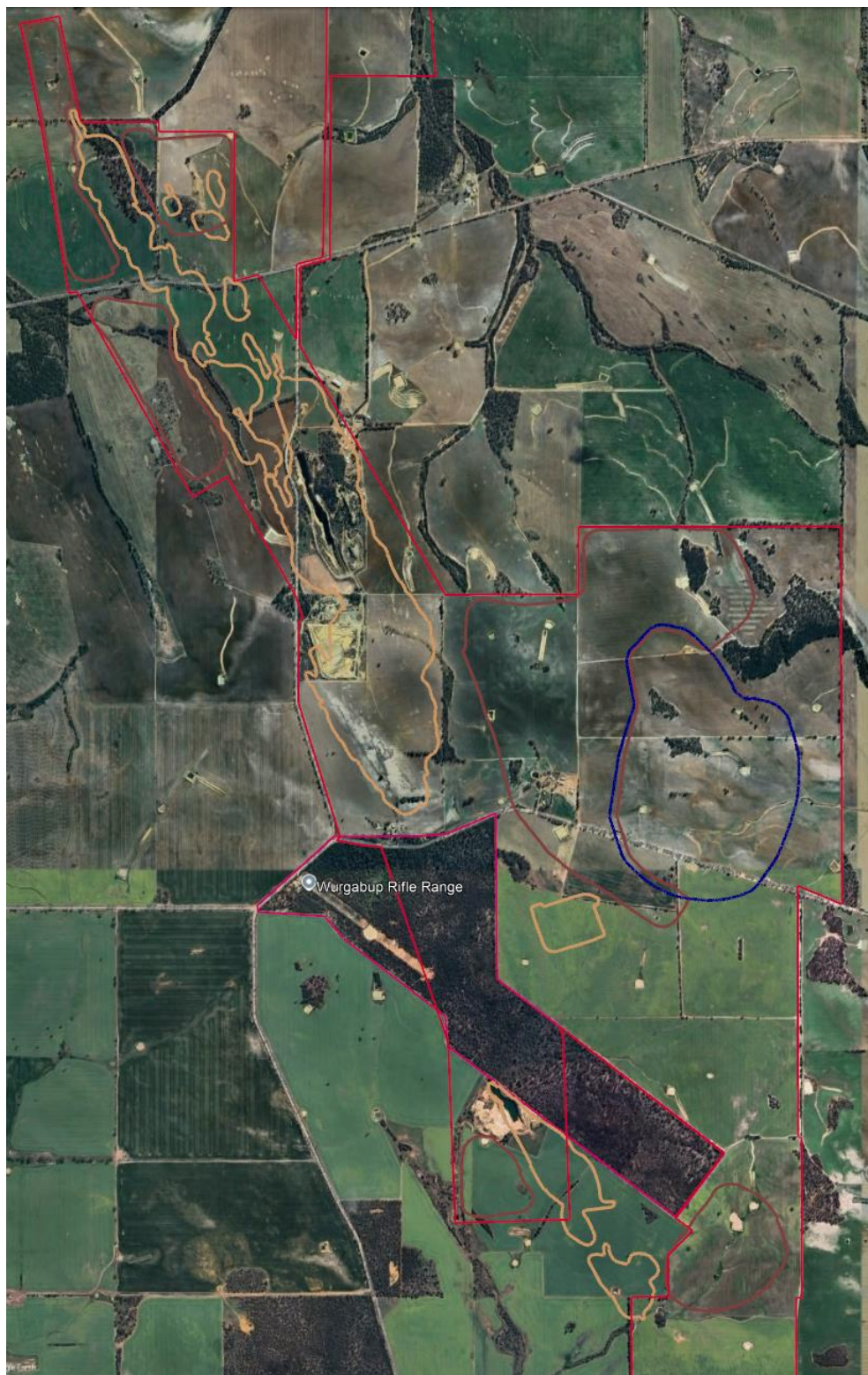
The general elevation of the proposed KGP open pit is 360 mRL  $\pm$ 20 m with the terrain sloping generally away from the pit at a gradient of less than 4%. Figure 2-8 shows the project area with 10 m contours of the topography and the tenement boundaries.

Figure 2-8 KGP Topography in 10m contours



Vegetation is sparse except for intermittent crops and corridors of trees along roadways. Figure 2-9 shows an aerial view of the KGP site with the approximate location of the proposed pits, waste dumps, TSF and ROM pad within the tenement boundaries.

Figure 2-9 Aerial view of KGP Site with approximate pit location



## 2.7 Climate

The climate of the Southwestern region of Western Australia is recognized as a regional steppe climate. The hot season lasts for 3.3 months, from 8 December to 16 March, with an average daily high temperature above 26°C. The hottest month of the year is January, with an average high of 30°C and low of 14°C. The cool season lasts for 3.6 months, from 24 May to 11 September, with an average daily high temperature below 17°C. The coldest month of the year is July, with an average low of 6.1°C and average high of 14.8°C.

The wetter season lasts 5.0 months, from 7 May to 6 October, with a greater than 16% chance of any given day being a wet day. The month with the most wet days is July, with an average of 10 days with at least 1 millimetre of precipitation. The drier season lasts 7.0 months, from 6 October to 7 May. The month with the fewest wet days in Katanning is February, with an average of 1.3 days with at least 1 millimetre of precipitation.

Table 2-1: Average Temperature Data for Katanning (source: [www.bom.gov.au](http://www.bom.gov.au))

Month	Lowest °C	Highest °C	Rainfall mm	Mean number of days above 1 mm
January	13.7	30.2	18.1	1.7
February	14.2	29.8	13.2	1.3
March	13.1	27.0	22.8	3.0
April	11.0	23.2	28.5	3.9
May	8.6	19.3	49.6	6.0
June	6.9	16.0	56.2	8.5
July	6.1	14.8	61.6	10.4
August	6.3	15.5	59.7	9.1
September	7.7	17.7	47.7	8.3
October	10.1	22.0	30.3	5.3
November	11.7	26.0	22.4	3.5
December	9.6	28.8	21.0	2.2

## 2.8 Conventions

### 2.8.1 Currency

All costs and prices are in Australian dollars (A\$) unless otherwise specified.

### 2.8.2 Units

The units of measure (volume, distance, etc.) used in this report, unless otherwise noted, are metric.

### 2.8.3 Mine Coordinates

All coordinates are reported in MGA84 zone 50H. This coordinate system is standard across the project to minimise errors when transferring data between study groups.

### 2.8.4 Block Dimensions

Three-dimensional entities in this report are described in the format X by Y by Z, where X refers to the Easting or across strike distance in metres; Y refers to the Northing or along strike distance in metres and Z refers to the RL or vertical distance in metres based on elevations above the Australian Height Datum.

## 2.9 Scope of Work

### 2.9.1 General

The scope of work for the DFS includes:

- Site visit to gain an understanding of the current state of the site.
- Request for Budget Pricing (RFBP) to obtain up-to-date open pit mine contracting costs.
- Open pit optimisation to explore the sensitivity of the new geological model to changes in key planning parameters.
- Detailed stage and ultimate designs for proposed open pit based on selected shells.
- Life of mine scheduling.
- Life of mine cost estimates.
- Report and associated ongoing deliverables.

### 2.9.2 Deliverables

Deliverables include:

- Mine planning model with depleted blocks for current topography and existing workings.
- Sign off sheets and optimisation cost models.
- Pit optimisation project including results/spreadsheets and DXF files.
- Pit designs in DXF and inventories reported in Excel.
- Life of mine schedules and spreadsheets for each option.
- Detailed cost models and spreadsheets for the final mine schedule option selected.
- Ongoing inputs to weekly meetings.
- Final report on Study findings and recommendations.

### 2.9.3 Exclusions

The scope is focused on the development of an open pit, and any future underground mine planning is excluded.

Metallurgical, geotechnical, hydrogeological, hydrology, environmental and other disciplines are undertaken by others.

## 3 MINING CONSIDERATIONS

### 3.1 Resource Model

The initial models provided had a parent cell size of 10x10x2.5 (XYZ) and were sub-celled to 5x5x2.5 (XYZ). Following review of these models, Orelogy requested a smaller sub-cell size to provide better resolution of the mineralisation as the ore zones at this block size were discontinuous in some area as illustrated in the cross sections shown in Figure 3-1 and Figure 3-2.

Orelogy uses edge dilution to model dilution and ore loss on a proportional basis within a larger block. To facilitate construction of the proportional blocks at a practical size, the starting block size needs to reflect the mineralisation boundaries and not a block aggregated to the smallest mining unit which already implies some dilution and ore loss is built in.

Figure 3-1 Initial Model Cross Section through Jackson & Olympia

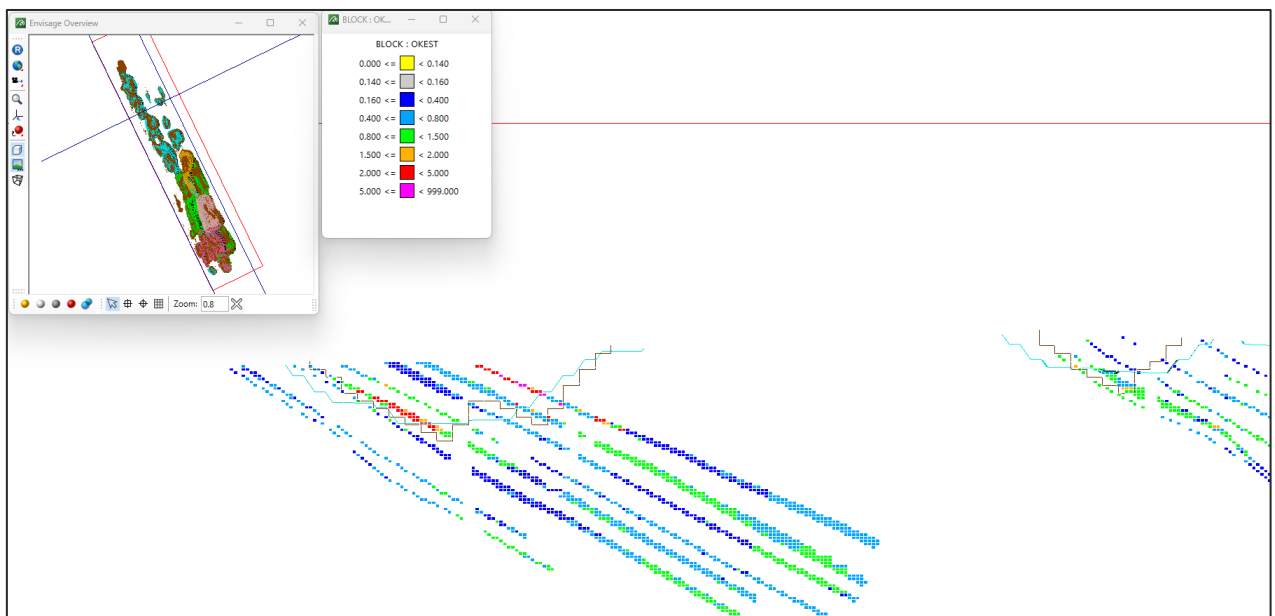
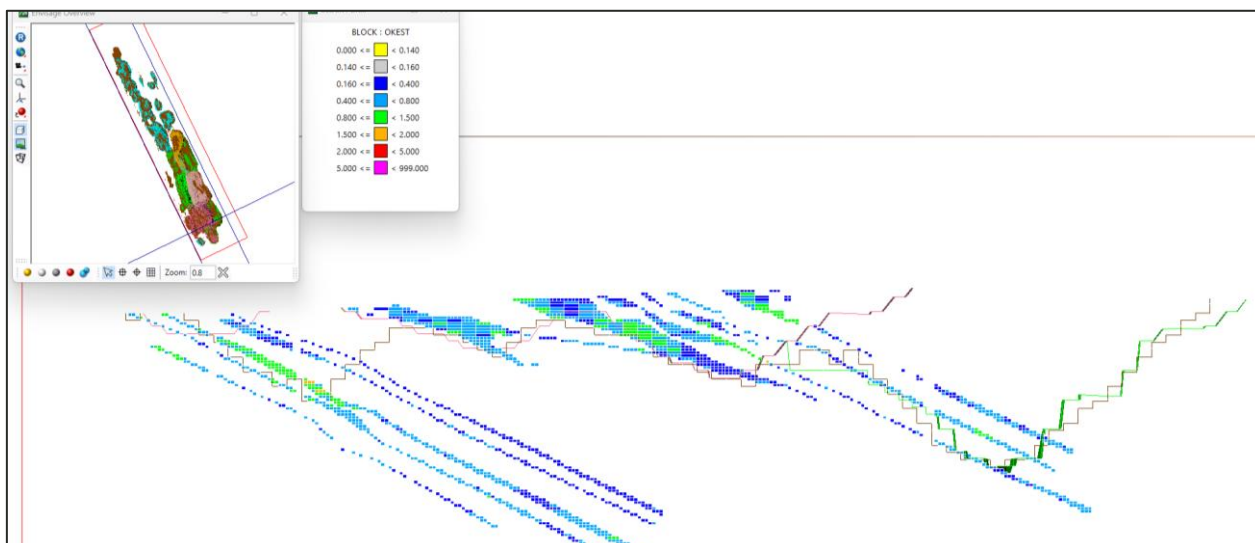


Figure 3-2 Initial Model Cross Section through Jinkas South



The final mineral resource models provided by SRK were in Datamine format and based on Local Uniform Conditioning at a block size of 2.5x2.5x2.5 with dimensions as highlighted in Table 3-1. The CZ model encompasses the proposed open pit mining area for Jinkas, Jinkas South, White Dam, Jackson, and Olympia, and is deep enough for open pit planning purposes. The SZ model covers a small area to the south for the Dingo pit. The NZ model covers an area more than 5km from the northern end of the CZ model. Table 3-2 summarises the models by resource category. The mine planning model also illustrates that 45% of the potential ore inventory is Measured, 34% is Indicated with the remainder of the inventory being in the Inferred category.

Table 3-1: Resource Model Characteristics

Detail		Central Zone	South Zone	North Zone
Model	Name	SMU_Central_0611.dm	SMU_Southern_0611.dm	SMU_Northern_0611.dm
Origin	Easting	584777.70	586,347.698	580,850
	Northing	6285993.31	6,281,343.314	6,294,850
	Level	-71.00	68.75	-100
Rotation	angle	-26	-26	0
Extent	Easting (m)	1,130	1,180	700
	Northing (m)	5,600	3,600	650
	Level (m)	560	325	525
Block Size	m	2.5	2.5	2.5
No. Blocks	X (m)	452	472	280
	Y (m)	2240	1440	260
	Z (m)	224	130	210

Table 3-2: Resource Tonnage by Model and Resource Category where Au  $\geq$ 0.45g/t, above 150mRL

Model	Resource Category	Density (t/m <sup>3</sup> )	Mass (kt)	Au Grade (g/t)	Gold (oz)
Central Zone	Measured	2.90	32,730	1.31	1,383
	Indicated	2.91	21,665	1.08	753
	Inferred	3.07	13,098	1.06	445
	<b>Sub-total</b>	2.93	67,493	1.19	2,581
South Zone	Measured	2.78	3,428	1.05	116
	Indicated	2.84	5,743	0.95	175
	Inferred	2.99	2,957	0.87	82
	<b>Sub-total</b>	2.86	12,128	0.96	372
North Zone	Measured	-	-	-	
	Indicated	2.70	422	1.62	22
	Inferred	2.80	812	1.36	35
	<b>Sub-total</b>	2.76	1234	1.45	57
Combined	Measured	2.89	36,158	1.29	1,498
	Indicated	2.89	27,830	1.06	949
	Inferred	3.04	16,867	1.04	563
	<b>Total</b>	2.92	80,855	1.16	3,011

## 3.2 Dilution and Ore Loss

A primary component of the mine planning cycle is the conversion of a resource block model to a diluted “mining model” that is suitable for mine planning purposes. There are a number of approaches currently utilised in the mining industry to model dilution, including but not limited to:

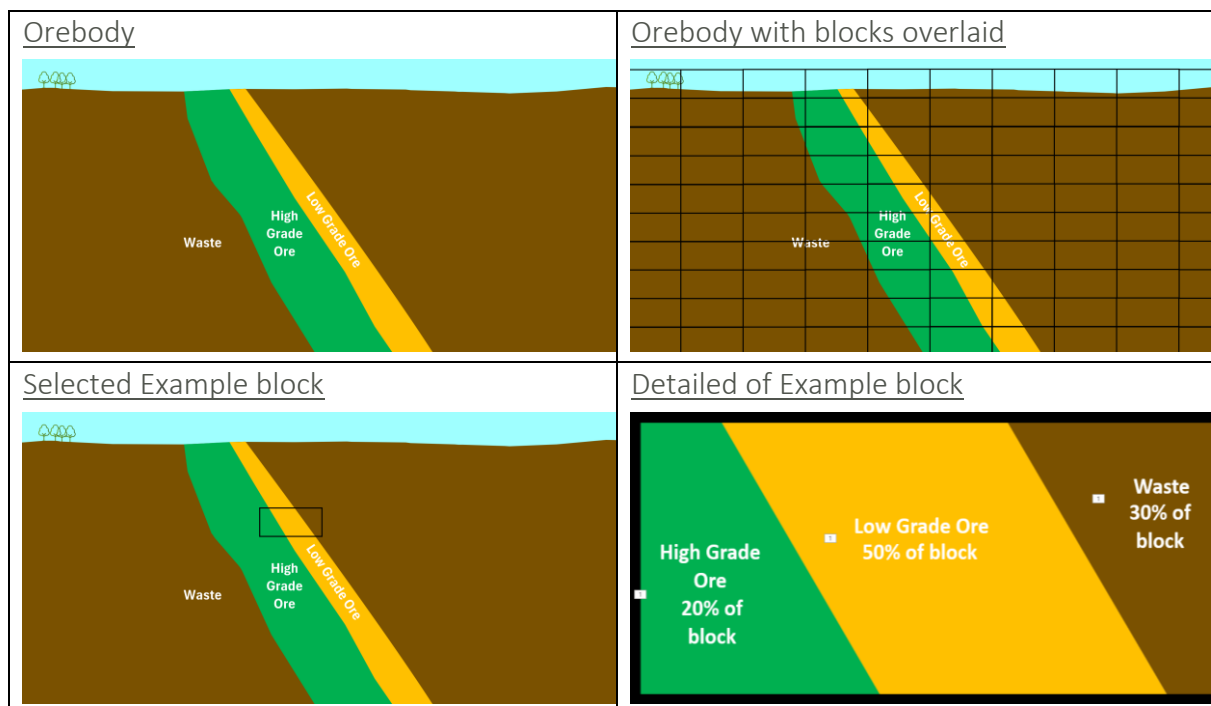
- Fixed Factors – i.e. the “Whittle approach”.
- Block Regularisation – Accumulating granular resource models in regular block sizes that replicate a Selective Mining Unit (SMU).
- Dilution skins or envelopes – Defining ore within a solid volume and then expanding that solid by a defined “skin width” and flagging the material captured as ore (i.e. diluent material).

Orelogy has developed an in-house proprietary approach that effectively models edge dilution in a similar fashion to the “skin” approach but does not require the generation of complex geometries and then re-modelling within them.

The Orelogy approach uses the following steps:

1. Regularises the resource model but maintains ore and waste proportions or “parcels” within the regular blocks. In this way the granularity of the sub-celled resource model is maintained within a regular block suitable for Whittle optimisation. This is shown graphically in Figure 3-3 below.

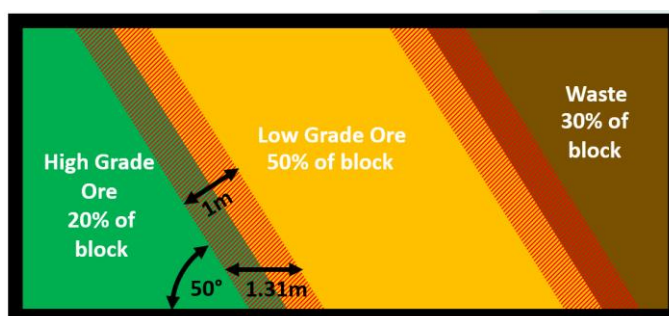
Figure 3-3 Regularisation Process to a Parcel Model



2. Determine a suitable “Zone of Mixing” between the different parcels based on:

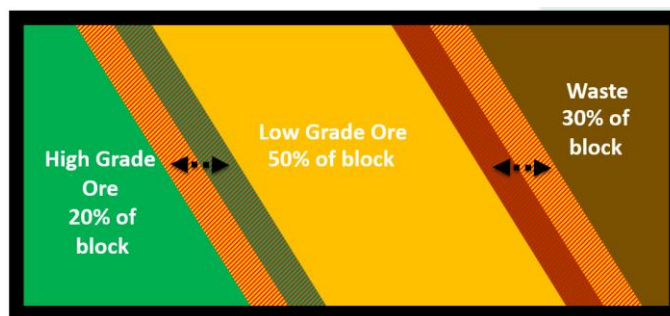
- Bench height
- Equipment size
- Degree of blasting movement (i.e. free dig through to high powder factor blasting)
- Dip of the orebody
- Refer to the example in Figure 3-4 below, based on:
  - True Mixing Width = 1 m
  - Orebody Dip = 50°
  - Apparent Width = 1 m / sin(50°) = 1.31 m

Figure 3-4 Determination of Zone of Mixing



3. The Zone of Mixing volume is then swapped out of each parcel and into the adjacent parcel as indicated in Figure 3-5.

Figure 3-5 Swapping of material within Mixing Zone



- o Consequently, the grades and densities associated with the swapped volumes are carried. This ensures:
  - all diluent grade is accounted for, including both value driving and / or deleterious elements.
  - the metal balance is maintained on a block-by-block basis

For the KGP, the SMU block size for dilution modelling was selected as 5 m E by 10 m N by 5 m RL. Two block across the ore zone in the vertical and horizontal directions provided enough resolution to find an edge. Twice this could be applied along strike as there was less variation in this direction and less edges likely.

Three different mix width regimes were modelled, assuming a bucket size of 2.0 m, as defined in Table 3-3.

Table 3-3 Apparent Mixing Widths

Parameter	Upside Case	Base Case	Downside Case
Digging tolerance as a percentage of bucket height	15%	20%	25%
True Mixing Width (rounded up to 0.1m)	0.3 m	0.4 m	0.5 m
Dip of orebody	35°	35°	35°
Apparent mixing width for script (rounded to 0.1m)	0.5 m	0.7 m	0.9 m

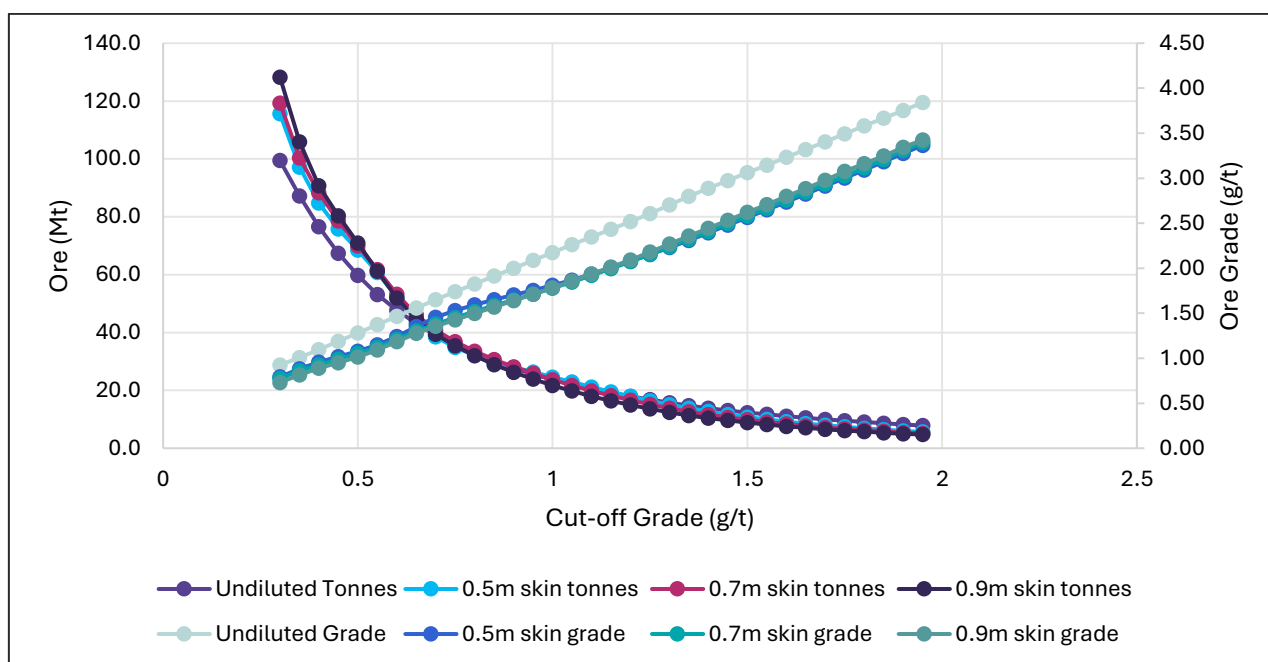
With the Orelogy approach replicating an edge effect:

- any blocks that are 100% ore are by implication within the orebody and therefore have no dilution applied.
- Any ore parcel where the block on either side are 100% waste is treated as an isolated block with two edges and the swapping volume is doubled.
- Any blocks where either the ore proportion or waste proportion are smaller than the mixing volume, they are insufficient in size to undertake the swap, and the two proportions are accumulated together.

Consequently, the degree of dilution varies on a block-by-block basis. Therefore, to report the global ore loss and dilution, the block dilution and ore loss is calculated and then these values are treated as a grade item and the weighted average can be reported.

The grade tonnage chart in Figure 3-6 shows that all three mixing widths reduce the resource grade by a similar amount with more tonnes added in the lower grade range for the larger mixing width.

Figure 3-6 Grade Tonnage curves for Undiluted and Diluted options



It is important to note that as the dilution varies locationally, different areas will generate different results. Reporting across an entire resource can be misleading as often an orebody becomes more broken up and spotty at the periphery of modelling. These areas are unlikely to form part of a reserve but will generate disproportionately high dilution which will skew the result. Therefore, it is important to report the global dilution numbers within either an optimisation shell or pit design to generate a result reflective of reality.

The results for the dilution modelling sensitivity are summarised in Table 3-4. The significantly higher dilution and ore loss for the downside case can be attributed to the narrowness of the ore zones, hence the need for selective mining practices.

Table 3-4 CZ Dilution modelling sensitivity

Mixing Zone Width	Upside Case 0.5 m	Base Case 0.7 m	Downside Case 0.9 m
Dilution	23.2%	26.1%	37.9%
Ore loss	20.5%	24.9%	31.5%

Table 3-5 shows details for the dilution modelling results of the 0.7 m mixing width case. Note that very few blocks for the SMU size were 100% ore.

Table 3-5: Dilution Modelling Results

Resource	Pre-dilution		Dilution material		Ore Loss		After Dilution		Overall		Undiluted Blocks (100% ore)
Category	Au (g/t)	Mt	Au (g/t)	Mt	Au (g/t)	Mt	Au (g/t)	Mt	Ore loss	Diln	
Unclassified	2.75	0.00	0.17	0.15	0.97	0.35	2.56	0.00	24.9%	26.1%	2.1%
Measured	1.17	29.93	0.22	5.19	0.87	6.32	1.05	28.46			
Indicated	1.21	4.67	0.18	1.44	1.03	1.95	1.02	4.31			
Inferred	1.10	0.19	0.18	0.04	0.92	0.06	0.92	0.18			
<b>CZ Total</b>	<b>1.18</b>	<b>34.80</b>	<b>0.21</b>	<b>6.83</b>	<b>0.91</b>	<b>8.67</b>	<b>1.05</b>	<b>32.95</b>			
Measured	1.10	1.59	0.20	0.24	0.89	0.26	1.00	1.57	16.2%	18.1%	5.0%
<b>SZ Total</b>	<b>1.10</b>	<b>1.59</b>	<b>0.20</b>	<b>0.24</b>	<b>0.89</b>	<b>0.26</b>	<b>1.00</b>	<b>1.57</b>			
Indicated	1.55	0.45	0.18	0.06	1.20	0.09	1.43	0.42			
Inferred	1.24	0.92	0.17	0.18	1.16	0.30	1.02	0.80	28.8%	24.8%	0.1%
<b>NZ Total</b>	<b>1.34</b>	<b>1.37</b>	<b>0.17</b>	<b>0.24</b>	<b>1.17</b>	<b>0.39</b>	<b>1.16</b>	<b>1.22</b>			

## 3.3 Water Management

### 3.3.1 Surface Water

Surface water generally flows away from the mining operation. A surface water management plan will be required to manage discharge of muddy or contaminated water from the site and diversion berms may be required to protect the pits from surface water flowing into the pits.

### 3.3.2 Ground Water

Advance dewatering of the pits using water bores is planned with water from the bores primarily be used for processing of the ore.

In-pit water management will primarily consist of runoff control and in-pit sumps. The pit catchment is approximately 60 hectares in area and the associated precipitation run-off will be controlled with the in-pit drainage water management system. It is assumed that the in-pit dewatering requirement is a maximum of 5 l/sec for the duration of the project.

## 3.4 Geotechnical

### 3.4.1 Rock Properties

The Geotechnical report compiled by 3<sup>rd</sup> Rock Consulting is based on 1,628 m of diamond core and test work including:

- Triaxial strength test of soils
- Hoek triaxial strength test of rock like material
- Uniaxial compressive Strength test of rock like materials
- Brazilian Tensile test of rock like material
- Direct Shear test of intact material and defect planes
- Particle size distribution of soil like materials
- Atterberg limits test for soil/clay materials
- Soil water characteristic curves of soil like materials
- Unsaturated Direct Shear test of soil like materials.

The result of the testing is summarised in Table 3-6 based on the weighted average for the deposits. In general, the Dingo deposit displays poorest rock properties and Jinkas displays the most competent fresh rock properties for the Quartz Monzonite.

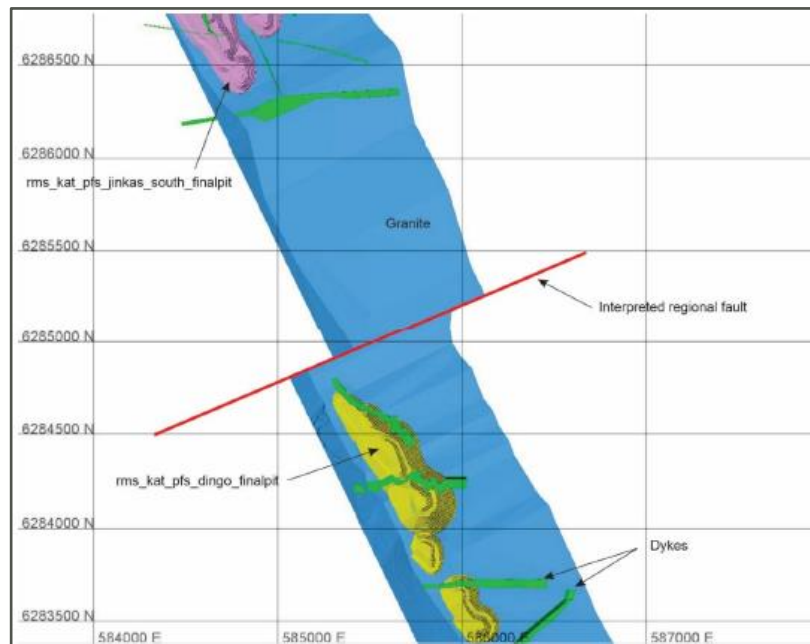
Table 3-6 Rock Mass Properties

Weathering	Rock Type	RMR	FF/m	Strength (UCS)	RQD
Oxide	Saprolite	30.8	28.7	2.7	2.2
Trans	Quartz Monzonite	54.9	5.3	44	54
	Granulite	42.7	17.7	51	34
Fresh	Quartz Monzonite	81.9	1.3	216	96
	Footwall Granite	81.7	1	184	100
	Mafic Dyke	69.5	2.7	101	95
	Granulite	78.7	3	228	93

### 3.4.2 Faulting

There is only one large-scale regional fault that is definitively known, offsetting the lithologies between the Jinkas South and Dingo pits, as shown in Figure 3-7. This fault has a strike of 67° and is assumed to be sub-vertical. It does not affect the Jinkas South or Dingo pits. Based on core logging and photographs, none of the drill holes are known to have intersected any major faults.

Figure 3-7: Current known Regional fault

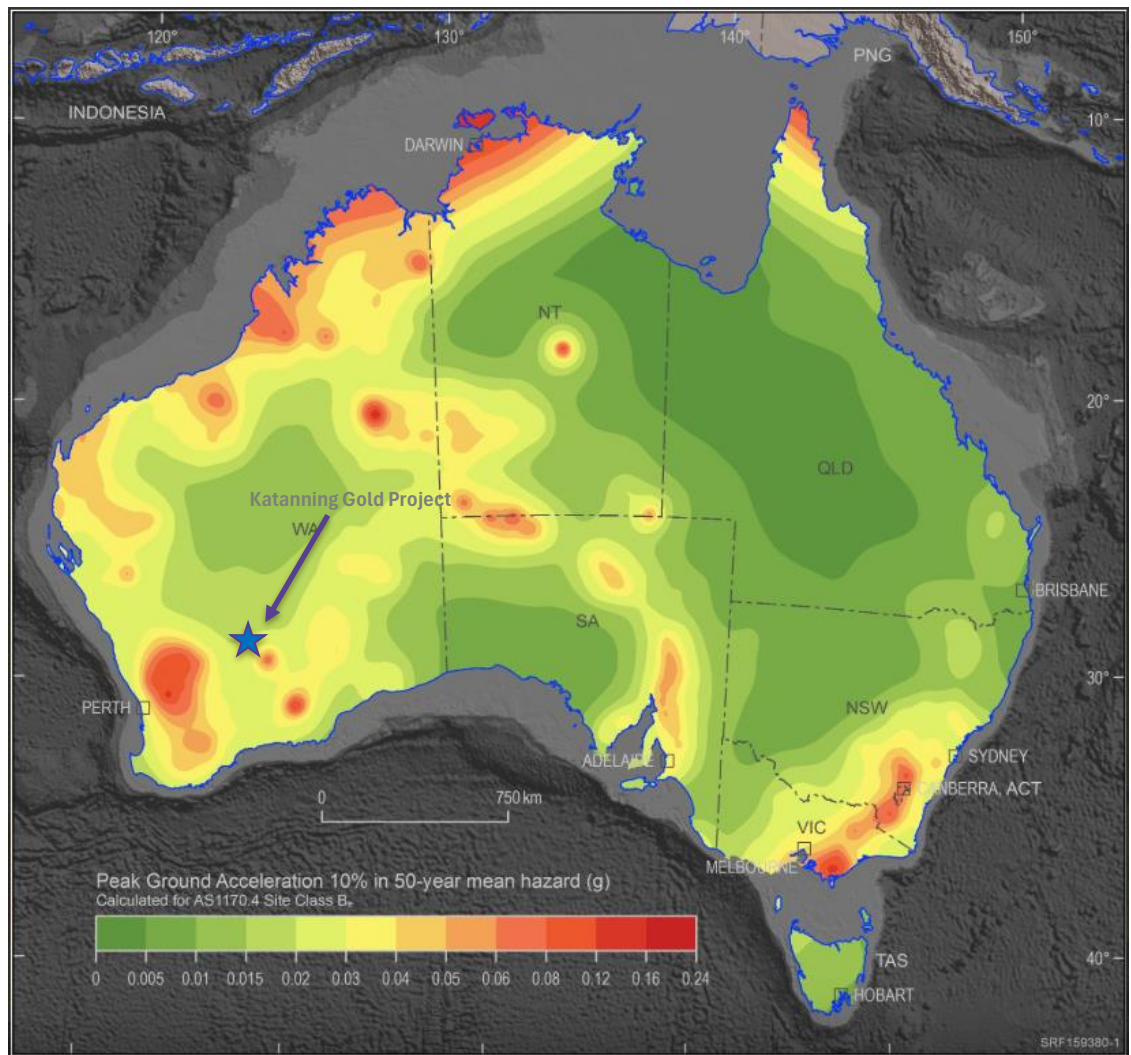


(source: 3RC Final DFS Geotechnical Report)

### 3.4.3 Seismicity

Geoscience Australia's 2018 Earthquake Hazard Map places the KGP in a moderate earthquake zone, with a 10% chance of exceeding a ground acceleration of 0.05 g over 50 years. To account for seismicity, a conservative seismic acceleration coefficient of 0.15 g, three times the published value for the area, was applied in slope stability analysis. This coefficient reflects higher seismic activity observed nearby.

Figure 3-8: NSHA23 hazard map indicating the mean PGA for 10% probability of exceedance in 50-years



(source: <https://www.ga.gov.au/scientific-topics/community-safety/projects/nsha>)

### 3.4.4 Slope Design

Open pit slope design criteria were provided by 3<sup>rd</sup> Rock Consulting as summarised in Table 3-7.

**Table 3-7: Open Pit Design Slope Design Parameters**

Deposit	Pit	Weathering	Lithology (slope dip direction)	Batter Height (m)	Berm Width (m)	Bench face angle	Inter-ramp slope angle
All	All	Saprolite	All	10	6.5	55	36.5
North	Jackson	Transitional	All	20	8.5	75	55.3
		Fresh	NF1 (225°-295°)	20	8.5	80	59.0
			NF2 (0°-225° & 295°-360°)	20	8.5	90	67.0
	White Dam	Transitional	All	20	8.5	75	55.3
		Fresh	CF1 (230°-260°)	20	8.5	80	59.0
			CF2 (0°-230° & 260°-360°)	20	8.5	90	67.0
	Jinkas Main & Jinkas South	Transitional	All	20	8.5	75	55.3
			CF1 (230°-260°)	20	8.5	85	59.0
			CF2 (0°-230° & 260°-360°)	20	8.5	90	67.0
	Dingo	Transitional	All	20	8.5	75	55.3
		Fresh	SF1 (230°-260°)	20	8.5	85	59.0
			SF2 (0°-230° & 260°-360°)	20	8.5	90	67.0

## 4 MINING METHODOLOGY

### 4.1 General

The mine operating strategy has been based on contract mining with Ausgold responsible for overall management, geology, grade control and mine planning aspects.

A mining contractor will be utilised for open pit mining as the preferred operational strategy for the following reasons:

- Reduces initial capital outlay and time requirements to build up and train a workforce.
- A contractor provides both equipment, personnel (locally sourced) with procedures and systems to reduce startup risk.
- Will generate the best project value with the greatest likelihood of success for a greenfields operation.

### 4.2 Mining Method

The preferred mining method at the KGP is to utilise open pit mining using conventional load and haul as this is a proven mining method for near surface low grade gold deposits, and the method is common practice in Western Australia. The operation is based on a mill throughput rate of approximately 3.6 Mtpa. The process plant is located to the south of the Central Zone (CZ) open pits, and east of the South Zone (SZ) pits.

The mining method and grade control practises to be employed at site are aimed at mining the ore zones selectively using backhoe configured excavators on a 2.5 m flitch to minimise dilution and ore loss.

Except for loose material at surface, all insitu material will require drill and blasting. The pits have been designed with berms at 10 m intervals in oxide and 20 m intervals for transitional rock and fresh rock. Blasting will be undertaken on 10 m benches for all material.

During the pre-production period, waste will be used for construction of the ROM pad, TSF embankment and other infrastructure, after which all waste will be hauled to the nearest waste dump or pit allocated for backfill.

Ore delivered to the ROM pad will be stored in stockpile 'fingers' from where it will be rehandled to the crusher using a Front End Loader (FEL) or FEL / truck combination.

All waste material from all stages will be hauled to waste rock dumps which will be constructed in 10 m lifts. Rehabilitation of the waste rock dumps will commence as soon as practicable.

### **4.3 Mine Infrastructure**

The existing infrastructure is located on the western boundary of the Katanning site is not suitable for the planned process plant but will be retained and used for ongoing exploration activities. The process plant has been relocated to the south of the Jinkas pit, east of the rifle range exclusion zone. The ROM is situated at the north end of the plant

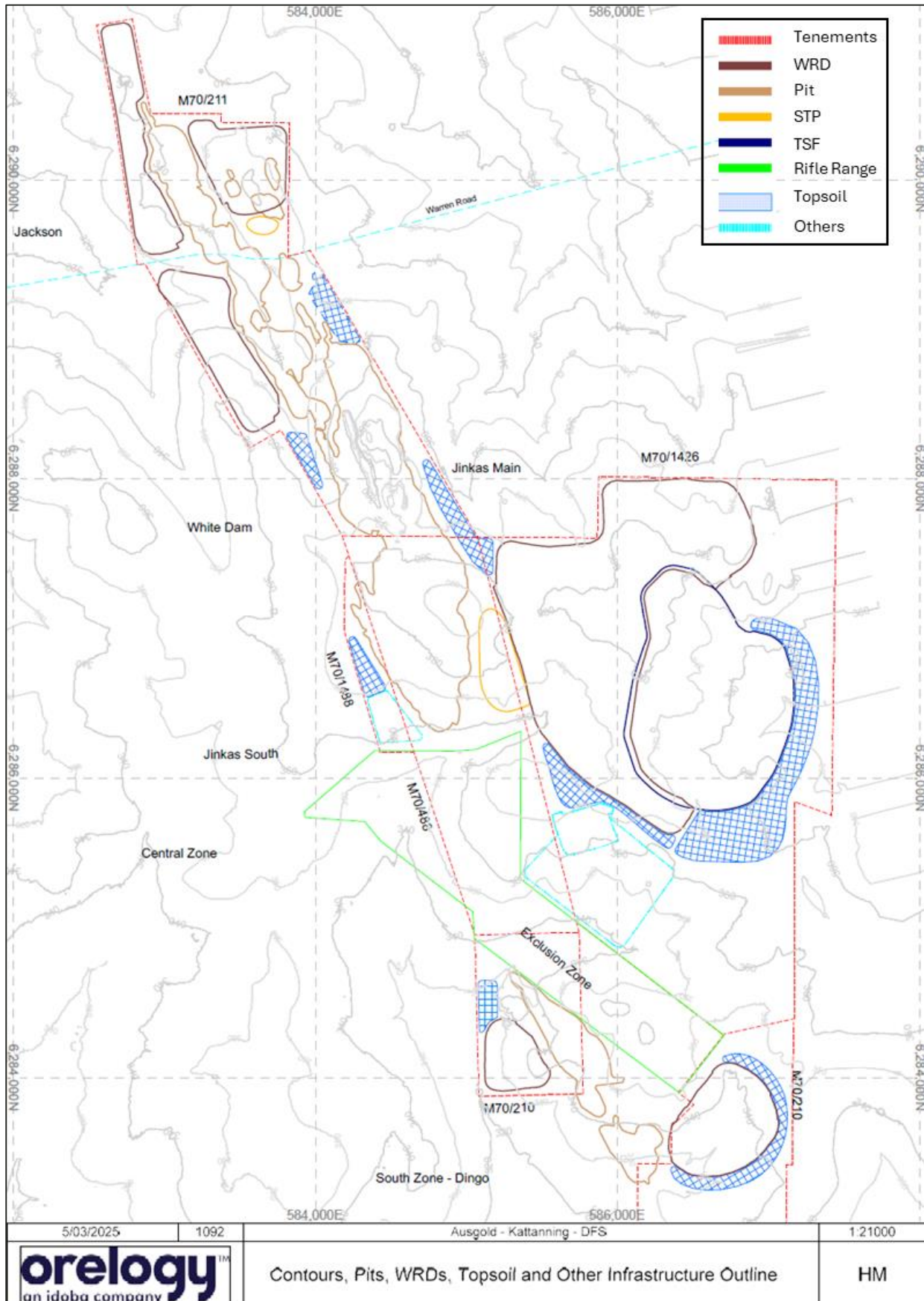
The overall site layout (of maximum disturbance) is shown in Figure 4-1 showing the location of the ROM pad, open pits, TSF, process plant and other infrastructure.

### **4.4 Clearing & Topsoil Removal**

Clearing of the moderately dense tree cover and light underbrush will be necessary before removal of topsoil to stockpiles for later use in rehabilitation of the site.

Topsoil generally has an average depth of 0.3 m and can extend to depths of 1.0 m. Topsoil dumps will be capped at 2.5 m high and will be placed on stockpiles around the periphery of the proposed waste dump. To preserve the soil viability and reduce rehandle costs, it is planned that rehabilitation is undertaken progressively. After initial topsoil stripping, any topsoil from new areas will be placed directly onto reprofiled waste dump slopes whenever possible.

Figure 4-1: Proposed Site Layout



## 4.5 Grade Control

Grade control drilling will be campaigned using reverse circulation (RC) methods to ensure sample quality. SRK completed an assessment for the drilling density and recommended a pattern at 7.5 m by 7.5 m at 20 m vertical deep intervals. Samples will be collected for 2.5 m composites using a 3-stage splitter to reduce sample size. Each sample will be bagged and labelled at the drill hole before being collected and sent to an on-site laboratory for assaying.

Initially, mining in each stage will comprise removal of overburden to a depth of 2 m to 5 m above the ore zone. From this level, each area will be quarantined for the grade control drilling campaign after which the ore will be marked out and mined selectively from the waste. Ore will be hauled to either the ROM pad located adjacent to the process plant, or to a long term stockpile for reclaim later in the mine life. The excavator operator will have access to the digital grade control plan and GPS location during mining.

## 4.6 Drill and Blast

All insitu material will be blasted in 10 m benches. The modified Kuzram method was used to assess the drill and blasting requirements for the material types expected at the KGP. The rock properties used for the assessment were extracted from the geotechnical report compiled by 3<sup>rd</sup> Rock Consulting as summarised in section 3.4.

Key assumptions were:

- Predominant rock types are:
  - Quartz Monzonite for waste zones
  - Granulite for ore zones.
- Joint conditions are “tight” for all weathering type
- Drilling precision would be managed through high precision GPS.
- Initiation would be controlled using electronic detonation systems.
- Bulk explosive would be an emulsion for use in wet conditions with a Relative Weight Strength (RWS) of 100 and density of 1.1 kg/L.
- Fragmentation targets of > 50% less than 250 mm and 97% less than 1.0 m.

For both ore and waste, higher energy explosives were found to deliver better results due to the fragmentation requirements. A drilling diameter of 172 mm was found to deliver the best results for waste rock with a smaller drilling diameter of 165 mm for ore.

Based on geotechnical guidance, buffer blasts and pre-shear/pre-split will be incorporated into the drill and blast processes to minimise damage to pit walls. It is assumed that buffer blasts will be free faced where possible, with lower powder factor with denser drill spacing than the production blasts.

The drill and blast design parameters are shown in Table 4-1.

**Table 4-1: Drill and Blast Design Parameters**

	Unit	Oxide Ore	Trans Ore	Fresh Ore	Oxide Waste	Trans Waste	Fresh Waste
Powder Factor	kg/bcm	0.45	0.50	1.00	0.45	0.55	1.00
Explosive Density	kg/L	1.1	1.1	1.1	1.1	1.1	1.1
Bench Height	m	10	10	10	10	10	10
Drill Size	mm	165	165	165	172	172	172
Burden	m	6.4	6.0	4.3	6.6	5.9	4.4
Spacing	m	7.4	6.9	4.9	7.6	6.8	5.1
Sub-drill	%	15%	15%	15%	15%	15%	15%
Stemming length	m	2.6	2.6	2.6	2.8	2.8	2.8
Charge weight	kg/hole	209	209	209	222	222	222

## 4.7 Load and Haul

The mine plan has been developed on the basis of 250 t class excavators for bulk waste with 120 t class excavators for selectively mining ore and associated waste in order to minimise dilution and ore loss.

As the ore dips at approximately 30° to the east, the ore will be faced up from the east hanging wall side to allow the waste to be stripped off with minimal ore loss. The bucket width of approximately 2.5 m for the 120 t class excavator will permit adequate selectivity when digging to the ore contact.

The 120 t class excavator has been matched to 90 t class rear dump trucks for ore mining, and the 250 t backhoe excavators matched to 140 t class trucks to excavate load and haul the bulk waste material as provides a better balance between physical number of pieces of mining equipment, mining selectivity and productivity.

## 4.8 Ancillary Services

The following ancillary equipment will be required for the activities detailed below:

- Dozers for floor control and waste dump tip-head management.
- Grader for road maintenance.
- Front end loader for ongoing ROM re-handle and stockpile reclaim.
- Front-end loader for general site maintenance.
- Water truck for dust suppression.
- Trailer mounted diesel pumps for dewatering sumps within the open pit.
- Lighting towers for night time operations.

## 4.9 Dewatering

Pits will be dewatered in advance using a network of water bores. Water extracted from the dewatering program will be piped to the raw water dam at the processing facility. Minor inflows into the pits will be the responsibility of the mining contractor with any hypersaline water optimally used for dust suppression.

## 4.10 ROM Management

The ROM operating strategy is based on stockpiling ore at a central location adjacent to the primary crusher. Blending for processing efficiency requires management of the oxide proportion in the feed to a maximum of 25% as well as “reactive” transitional ore to a maximum of 5%.

The ore excavator, with a capacity to mine approximately 900 dry metric tonnes (dmt) per hour, can mine ore at almost double the rate at which the plant can process the ore (520 dmt/hr for oxide and 450 dmt/hr for trans/fresh). When taking into consideration the rehandle of the other 50% of the ore, and timing window for excavator and crusher operating simultaneously, there is less than 15% opportunity to direct tip the ore into the crusher. Therefore, all ore is assumed to be fed from a finger stockpile located on the ROM pad by a FEL.

## 5 Open Pit Optimisation

### 5.1 Open Pit Optimisation Overview

The first stage of the conversion of a mineral resource into a mineable open pit reserve is the open pit optimisation process. It is at this stage that all the latest physical, technical and economic parameters are applied to the orebody to generate the “ideal” open pit excavation geometry.

The WHITTLE™ open pit optimisation software tool was utilised by Oreology to undertake this component of the study. WHITTLE™ is a recognised, off the shelf, optimisation package, and to a degree is considered the industry standard for open pit optimisation.

### 5.2 Open Pit Optimisation Methodology

In broad terms, the process that WHITTLE™ undertakes is to vary the base input price by a range of factors (referred to as the Revenue Factors), up and down from a base value of 1. For any given Revenue Factor WHITTLE™ then produces a three-dimensional shape, or “shell”, that generates the maximum possible value for all the input parameters and the associated factored price.

Lower factored prices will produce smaller shells; the higher price factor, the larger the shell. This results in a set of “nested” shells, with each shell lying inside the shell of the next largest Revenue Factor.

The value of each shell is reported at the base price. The effect of this is that:

- Shells with a Revenue Factor  $<1$  are smaller and have less ore than the shell with Revenue Factor 1. This reduces the revenue generated and therefore they will have a lower value.
- Shells with a Revenue Factor  $>1$  are bigger and have relatively more waste. This increases the costs and therefore they will have a lower value.
- The result is the classic WHITTLE™ cash flow curve that generally peaks at the base price (i.e., Revenue Factor 1). The robustness of this shell is reflected in how quickly the value falls away either side of this peak.

These nested shells are important for several reasons:

- The smaller shells indicate the areas of highest value in the ore body and therefore give a guide as to where mining should commence.
- The larger shells provide an indication of how much additional mineralisation may become economic, or alternatively what current ore may become unviable, should input parameters change in the future.

- They permit WHITTLE<sup>™</sup> to develop a “schedule” for mining the deposit over time and therefore allow a discounted cash flow (DCF) to be generated.

It is important to have some appreciation of how WHITTLE<sup>™</sup> generates a DCF as these values are used in this study as a guide to both sensitivity and the shell selected as the basis for pit design. It should be noted that WHITTLE<sup>™</sup> cash flows differ from accounting cash flows which include the effects of debt servicing, depreciation and taxation.

WHITTLE<sup>™</sup> generates the following two standard DCF's:

**Worst Case DCF** assumes that, for any given shell, extraction is undertaken bench by bench sequentially from the top to bottom of the pit. This results in a significant amount of any overlying overburden being removed prior to presentation of ore (i.e., a large pre-strip), and there are no initial shells or stages that access higher value ore earlier in the schedule. This is clearly not a preferred pit extraction scenario from a value perspective.

**Best Case DCF** assumes that, for any given shell, extraction is undertaken sequentially from the smallest shell generated by the optimisation, through all the intervening shells out to the shell selected. This approach focuses on the highest value areas of the orebody and generates the least amount of waste stripping. As such it provides the highest DCF for the selected shell. However, this approach is constrained by practical considerations as it is effectively mining the orebody using an impractically high number of small incremental pushbacks that could not be achieved in reality as:

- The distances between successive shells are invariably too narrow to mine practically.
- The bench turnover required to mine the shell in the time frame is not achievable.
- The base of the shell is too small and constricted to allow for practical access or safe extraction.

Figure 5-1 and Figure 5-2 illustrates a schematic representation of the Worst Case DCF mining and Best Case DCF mining sequences. The figures indicate the amount of waste that needs to be moved to gain access to the orebody, and as can be seen the Worst Case approach requires significantly more waste movement than the Best Case, hence two divergent cash flows are generated.

In essence, these two DCF'S provide the extremities or theoretical value that can be generated from a deposit. The shell that represents what will happen in reality will in fact lie somewhere between these two endpoints.

Figure 5-1: Worst Case DCF Mining Sequence



Figure 5-2: Best Case DCF Mining Sequence



Figure 5-1: Worst Case DCF Mining Sequence



Figure 5-2: Best Case DCF Mining Sequence



## 5.3 Inputs and Assumptions

Only Measured and Indicated resource categories were included in the pit optimisation runs used to define the economic shell selected for pit design. The Inferred category was included in alternative options to help define the potential extent of future mining activity for placement of infrastructure.

All blocks in the model were depleted based on the current as-mined and backfilled topography.

### 5.3.1 Gold Price

At the commencement of the mining study, the gold price was set at \$3,000/oz with sensitivities at \$2,400/oz, \$2,700/oz, \$3,300/oz, \$3,600/oz and \$4,200/oz.

### 5.3.2 Selling Costs

An average royalty rate of 2.5% with a refining cost of \$7.50/oz were included in the selling costs.

### 5.3.3 Wall Slope Parameters

Geotechnical guidance provided berm and batter configurations including inter-ramp slope angles (refer to Figure 5-3) by face angle for all weathering domains. Orelogy applied ramp allowances as

outlined in Table 5-1. The ramp allowances assumed the number of ramps crossing each wall. For example, the oxide ramp allowance of 3 m is based on a 30 m wide ramp of 100 m in length for the 10 m bench height at 1000 m intervals around the pit crest.

Figure 5-3: Slope Design Geometrics

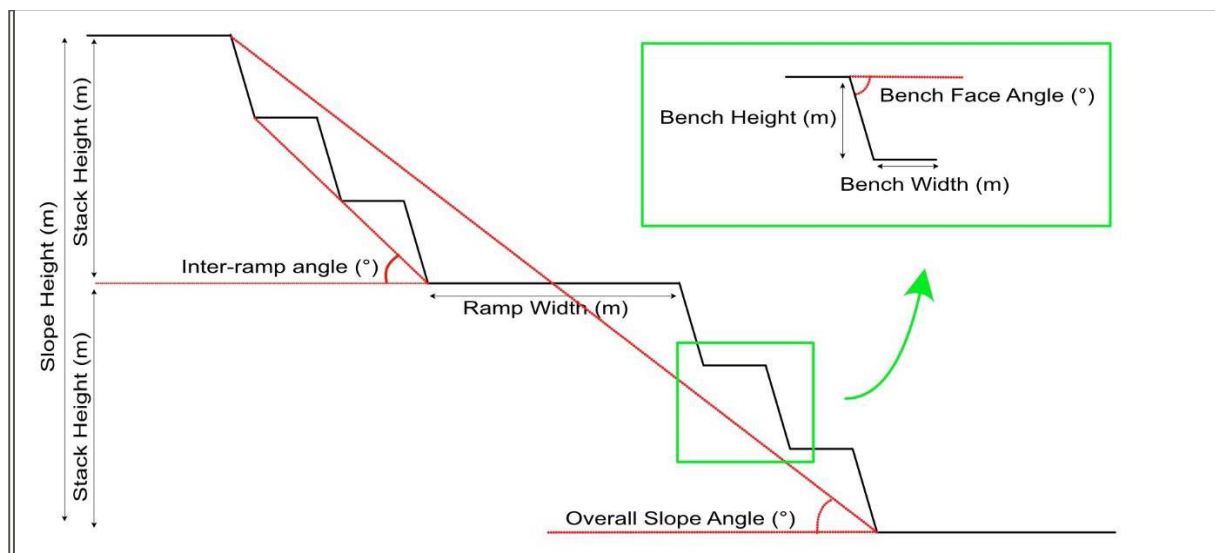


Table 5-1: Geotechnical Parameters for Pit Optimisation and Mine Design

Area	Domain	Berm Height	Berm Width	Batter Angle	Approx Wall Height	Ramp Allowance	Overall Slope Angle
		m	m	°	m	m	°
All	Oxide	10	6.5	55	20	3	40.4
All	Trans	10	6.5	80	40	6	51.3
Jackson & Olympia	NF1 Fresh	20	8.5	80	40	15	52.6
	NF2 Fresh	20	8.5	90	40	18	56.5
White Dam, Jinkas & Jinkas South	CF1 Fresh	20	8.5	85	150	30	56.7
	CF2 Fresh	20	8.5	90	150	46	56.0
Dingo	SF1 Fresh	20	8.5	85	40	15	56.0
	SF2 Fresh	20	8.5	90	40	30	46.1

### 5.3.4 Mining Costs

All waste mining costs were coded into the mining model for each bench based on a regression which was used to fit the mining costs from a limited dataset across all benches. These mining costs were developed from the PFS mining contractor submissions and include load, haul, drill, blast functions together with the ancillary mining fleet and fixed mining costs. These costs are illustrated in Figure 5-4 and in Table 5-2.

Figure 5-4: Mining Costs by Level

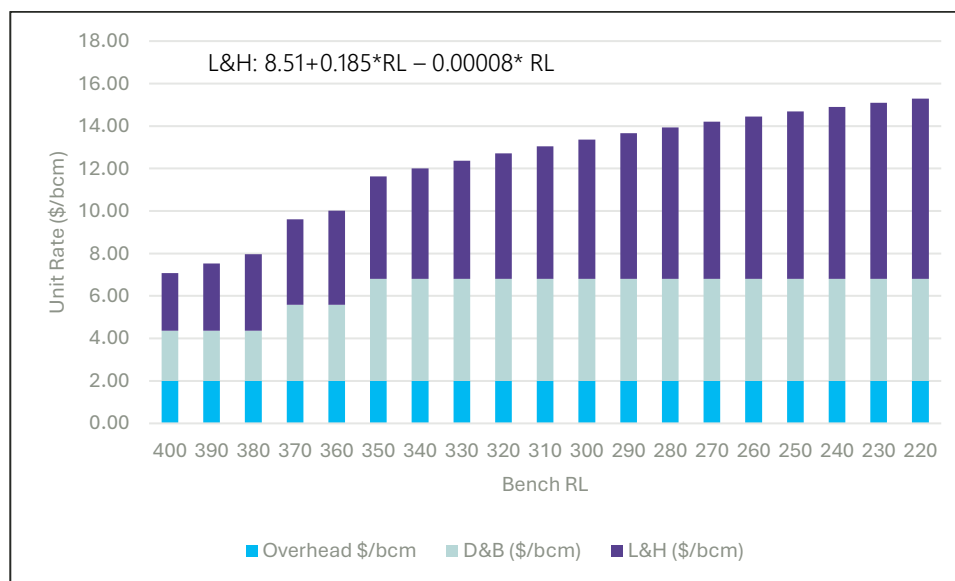


Table 5-2: Pit Optimisation Parameters

Metres below surface		\$/bcm	D&B \$/bcm	Fixed O'heads \$/bcm	WASTE Total	
from	to				\$/bcm	\$/t
Surface	390	2.71	2.37	1.99	7.07	3.93
390	380	3.17	2.37	1.99	7.53	4.18
380	370	3.61	3.59	1.99	7.97	4.43
370	360	4.03	3.59	1.99	9.62	3.55
360	350	4.44	3.59	1.99	10.02	3.70
350	340	4.84	4.81	1.99	11.64	4.14
340	320	5.21	4.81	1.99	12.01	4.28
320	310	5.57	4.81	1.99	12.38	4.40
310	300	5.92	4.81	1.99	12.72	4.53
300	290	6.25	4.81	1.99	13.05	4.64
290	270	6.56	4.81	1.99	13.36	4.76
270	260	6.86	4.81	1.99	13.66	4.86
260	250	7.14	4.81	1.99	13.94	4.96
250	240	7.40	4.81	1.99	14.21	5.06
240	230	7.65	4.81	1.99	14.45	5.14

### 5.3.5 Processing Inputs

Ore processing costs were provided by MineScope Services dependent on throughput rates and degree of weathering for the various pits. Grade control and ore rehandle costs were added by Orelogy to provide a total processing cost adjustment factor for each ore type that can be processed as outlined in Table 5-3.

Table 5-3: Process unit costs

Domain	Pit	Throughput rate	Variable process cost	Fixed Process cost	Ore GC & Rehandle costs	Total PCAF
		Tph	\$/t	\$/hr	\$/t	\$/t
Oxide	All	518	13.39	6,134	2.55	<b>27.79</b>
Transitional	Jinkas, White Dam, Olympia, Dingo	476	15.33	6,134	2.55	<b>30.77</b>
	Jackson	476	17.65	6,134	2.55	<b>33.09</b>
Fresh	Jinkas, White Dam, Olympia	450	14.33	6,134	2.55	<b>30.51</b>
	Jackson	450	15.62	6,134	2.55	<b>31.81</b>
	Dingo	424	14.33	6,134	2.55	<b>31.34</b>

### 5.3.6 Process Recoveries & Breakeven cut-off grade

Breakeven cut-off grade can be calculated using the formula:

$$COG = \text{Process cost} / (\text{Recovery} \times \text{Net Price})$$

MineScope Services advised that recovery was dependent on the tail grade and provided head grade based formula for the tail grade by degree of weathering for the various pits, similar to the process costs. A maximum recovery was also supplied.

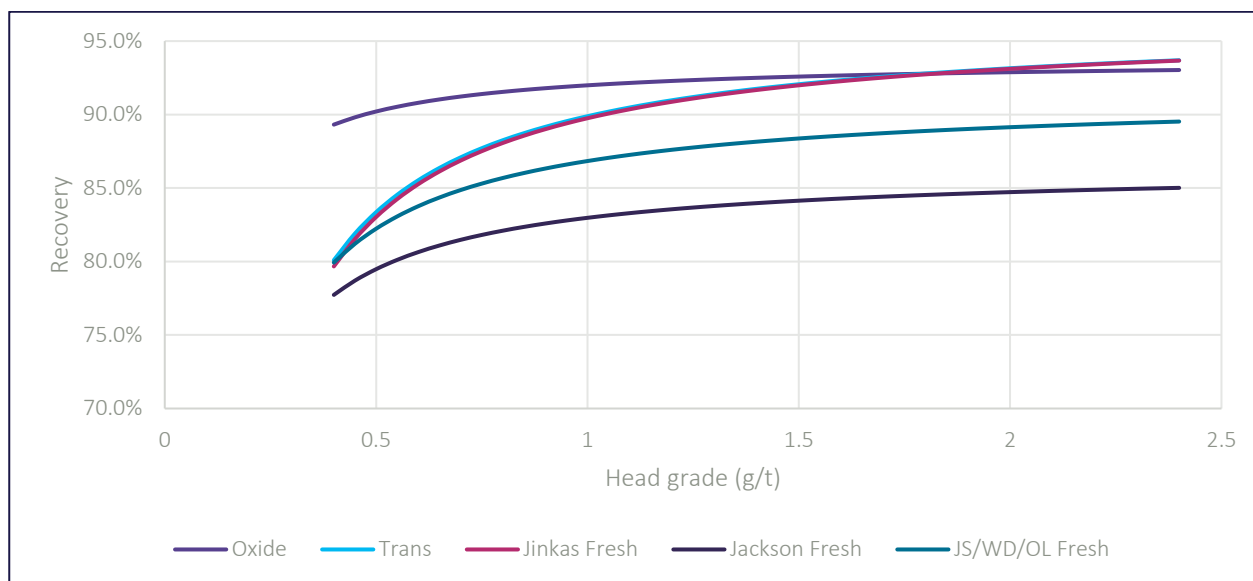
As the recovery is dependent on the head grade, it is possible to iteratively solve the lower end of the recovery range for the breakeven cut-off grade as per Table 5-4. The recovery curves by head grade are shown in Figure 5-5.

$$\text{Recovery} = 1 - (\text{Tail grade}) / (\text{Head Grade}), \text{ where Tail grade} = m \times \text{Head grade} + b.$$

Table 5-4 Tailing Grade Based Recovery

Domain	Pit	m	b	Max Recovery	Breakeven COG	Calculated Minimum Recovery
				%	g/t	%
Oxide	All	0.062	0.018	96	0.335	<b>88.5</b>
Transitional	Jinkas, White Dam, Olympia & Dingo	0.036	0.065	95	0.408	<b>80.4</b>
	Jackson	0.036	0.065	95	0.434	<b>81.4</b>
Fresh	Jinkas	0.035	0.067	95	0.407	<b>80.0</b>
	Jackson	0.135	0.035	95	0.433	<b>78.4</b>
	Dingo	0.072	0.053	95	0.417	<b>80.0</b>
	Jinkas South, White Dam & Olympia	0.086	0.046	95	0.410	<b>80.2</b>

Figure 5-5 Recovery Curves



### 5.3.7 Capital Costs Inputs

Capital costs are not included as part of the optimisation process as they do not affect shell selection.

## 5.4 Optimisation Results

### 5.4.1 Central Zone

A range of optimisation runs were considered and focused on changes to key parameters including:

- Resource category
- Mining costs
- Processing costs
- Gold price
- Overall slopes

Initially, the CZ MRE block model was imported to Whittle and optimised on all resource categories without any constraints to establish the possible extents of the open pit. The diluted model was then imported and run using the same parameters before additional runs adding tenement constraints, excluding Inferred material and finally applying a minimum grade cut-off of 0.4 g/t. The changes on pit size, contained ore and recoverable ounces were plotted as the waterfall charts presented in Figure 5-6. The waterfall charts show that dilution has the greatest impact on the models followed by the tenement constraints with minimal effect from applying a 0.4 g/t cut-off or excluding inferred material.

All runs for the Central Zone are summarised in Table 5-5 with the Base case (SC103) highlighted.

Figure 5-6 Waterfall charts for Optimisation Runs Focusing on Sensitivity to Constraints

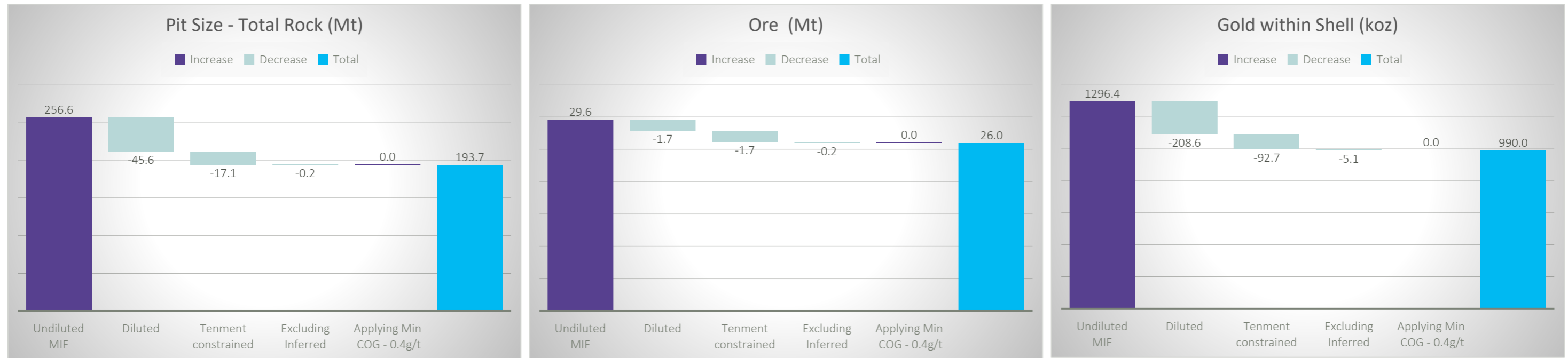


Table 5-5: Central Zone Pit Optimisation Runs Focusing on Sensitivity to Various Parameters

Scenario	Description	Material						Financials						Mine Life	Strip Ratio	% Inferred
		Ore Mined		Waste	Total	Ounces	Rec Ounces	Undiscounted			Cost per Oz					
		TOTAL ORE	Au g/t					Mining Cost	Processing Cost	Selling Costs		Revenue	Cashflow			
		TOTAL Mt	Au g/t	Mt	Mt	koz	koz	\$M	\$M	\$M	\$M	\$M	\$/oz			
<b>WATERFALL</b>																
<a href="#">SCN 106</a>	Diluted - within Tenements - no COG - M&I, 3000 AUD/oz	26.0	1.19	167.8	193.7	990	895	-\$886.4	-\$796.9	\$0.0	\$2,610.2	\$926.8	-\$1,882	7.4	6.5	0%
<a href="#">SCN 107</a>	Diluted - within Tenements - MIF, 3000 AUD/oz	26.1	1.18	167.8	193.9	995	899	-\$887.4	-\$801.7	\$0.0	\$2,623.6	\$934.5	-\$1,878	7.4	6.4	1%
<a href="#">SCN 108</a>	Diluted - No tenement constraint - MIF, 3000 AUD/oz	27.9	1.21	183.2	211.0	1,088	985	-\$971.0	-\$855.7	\$0.0	\$2,872.9	\$1,046.2	-\$1,855	8.1	6.6	1%
<a href="#">SCN 109</a>	Undiluted - No tenement constraint - MIF, 3000 AUD/oz	29.6	1.36	227.0	256.6	1,296	1,178	-\$1,183.9	-\$909.3	\$0.0	\$3,436.1	\$1,342.9	-\$1,777	9.2	7.7	1%
<b>SENSITIVITIES</b>																
<a href="#">SCN 121</a>	Processing -20% - M&I only, 3000 AUD/oz	28.5	1.15	182.7	211.1	1,050	947	-\$966.3	-\$701.5	\$0.0	\$2,761.5	\$1,093.7	-\$1,762	8.3	6.4	0%
<a href="#">SCN 122</a>	Processing -10% - M&I only, 3000 AUD/oz	27.4	1.16	173.8	201.2	1,020	921	-\$920.4	-\$757.6	\$0.0	\$2,686.6	\$1,008.7	-\$1,822	7.9	6.3	0%
<a href="#">SCN 123</a>	Processing +10% - M&I only, 3000 AUD/oz	23.6	1.23	156.0	179.5	931	845	-\$820.6	-\$792.6	\$0.0	\$2,465.4	\$852.2	-\$1,909	6.9	6.6	0%
<a href="#">SCN 124</a>	Processing +20% - M&I only, 3000 AUD/oz	21.5	1.28	150.2	171.7	887	807	-\$784.2	-\$787.7	\$0.0	\$2,355.9	\$783.9	-\$1,947	6.5	7.0	0%
<a href="#">SCN 125</a>	Mining -20% - M&I only, 3000 AUD/oz	29.3	1.15	197.5	226.8	1,084	977	-\$831.7	-\$900.6	\$0.0	\$2,849.3	\$1,117.1	-\$1,774	8.7	6.7	0%
<a href="#">SCN 126</a>	Mining -10% - M&I only, 2850 AUD/oz	27.2	1.17	178.5	205.7	1,026	926	-\$847.1	-\$836.1	\$0.0	\$2,701.7	\$1,018.5	-\$1,818	7.8	6.6	0%
<a href="#">SCN 127</a>	Mining +10% - M&I only, 3000 AUD/oz	24.6	1.19	152.8	177.4	942	853	-\$892.0	-\$754.1	\$0.0	\$2,489.6	\$843.6	-\$1,929	6.9	6.2	0%
<a href="#">SCN 128</a>	Mining +20% - M&I only, 3000 AUD/oz	22.4	1.21	133.4	155.8	870	789	-\$849.0	-\$685.7	\$0.0	\$2,302.0	\$767.4	-\$1,945	6.3	6.0	0%
<a href="#">SCN 129</a>	Dilution 0.5m skin (-25%) - M&I only, 3000 AUD/oz	24.8	1.22	166.1	190.9	975	883	-\$873.3	-\$761.1	\$0.0	\$2,577.3	\$943.0	-\$1,850	7.0	6.7	0%
<a href="#">SCN 130</a>	Dilution 0.9m skin (+25%) - M&I only, 3000 AUD/oz	25.6	1.15	154.4	180.0	946	855	-\$822.9	-\$784.2	\$0.0	\$2,494.6	\$887.5	-\$1,880	7.1	6.0	0%
<a href="#">SCN 131</a>	Steeper Slopes -10% - M&I only, 3000 AUD/oz	27.2	1.20	165.3	192.5	1,052	951	-\$882.0	-\$834.8	\$0.0	\$2,774.7	\$1,057.9	-\$1,805	7.9	6.1	0%
<a href="#">SCN 132</a>	Flatter slopes +10% - M&I only, 3000 AUD/oz	22.3	1.14	139.5	161.8	820	741	-\$728.9	-\$682.5	\$0.0	\$2,161.5	\$750.1	-\$1,905	6.3	6.3	0%
<a href="#">SCN 133</a>	Minimal ramp allowance - M&I only, 3000 AUD/oz	32.4	1.14	207.1	239.4	1,190	1,072	-\$1,103.9	-\$994.3	\$0.0	\$3,127.7	\$1,029.5	-\$1,957	9.2	6.4	0%
<b>PRICE</b>																
<a href="#">SCN 101</a>	Diluted - Tenement constrained - 0.4g/t Cut- M&I, 2400 AUD/oz	15.4	1.41	103.1	118.5	697	637	-\$538.6	-\$470.8	\$0.0	\$1,486.6	\$477.2	-\$1,584	4.7	6.7	0%
<a href="#">SCN 102</a>	Diluted - Tenement constrained - 0.4g/t Cut- M&I, 2700 AUD/oz	20.4	1.27	130.9	151.3	834	759	-\$686.5	-\$624.5	\$0.0	\$1,991.5	\$680.4	-\$1,728	5.8	6.4	0%
<a href="#">SCN 103</a>	Diluted - Tenement constrained - 0.4g/t Cut- M&I, 3000 AUD/oz	26.0	1.19	167.8	193.7	990	895	-\$886.4	-\$796.9	\$0.0	\$2,610.2	\$926.8	-\$1,882	7.4	6.5	0%
<a href="#">SCN 104</a>	Diluted - Tenement constrained - 0.4g/t Cut - M&I, 3300 AUD/oz	28.9	1.14	186.5	215.4	1,061	957	-\$985.7	-\$887.5	\$0.0	\$3,070.5	\$1,197.3	-\$1,958	8.2	6.5	0%
<a href="#">SCN 105</a>	Diluted - Tenement constrained - 0.4g/t Cut - MIF, 3600 AUD/oz	32.5	1.11	217.4	249.9	1,156	1,039	-\$1,151.7	-\$999.3	\$0.0	\$3,640.0	\$1,488.9	-\$2,070	9.3	6.7	0%

Figure 5-7 presents the sensitivity analysis for the Central Zone which shows that resource is most sensitive to changes in gold price with more downside from lower prices than upside from higher prices. Observations are:

- While price has the largest impact on ore tonnes and ounces resulting in significant changes to DCF, it has little to no impact on strip ratio.
- Conversely, the dilution skin has the largest effect in strip ratio with low impact on ore tonnes, ounces or DCF.
- Mining costs and processing costs have similar effects on the ore tonnes, ounces and DCF with opposite effects on strip ratio where higher mining costs result in a lower strip ratio and higher processing costs result in a higher strip ratio.

**Figure 5-7: Central Zone Pit Optimisation Sensitivity Runs (showing changes with ±10% variance in inputs)**



### 5.4.2 South Zone

As with the Central Zone, the South Zone was initially optimised on the MRE model without constraints. Subsequent runs assess the effects of dilution, exclusion of Inferred material, mining tenement constraints, and the woodland exclusion zone.

The unconstrained shells extended outside the current mining lease (ML) into the adjacent exploration lease (EL) E70/2928 and into the adjacent Rifle range woodland. The mining tenement constraint was found to be a significant factor in the viability of the shell; however, it is reasonable

to assume that Ausgold will be able to convert part of the EL to a ML before this area is mined, the tenement constraint was removed.

This is not the case for the adjacent Rifle range woodland (as highlighted in Figure 4-1) which is currently protected. Therefore, SC206 was selected as the base case as it excludes the Woodland.

All runs for the South Zone are summarised in Table 5-6 with the Base case (SC206) highlighted.

### 5.4.3 North Zone

Optimisation for the North zone commenced with the undiluted MRE model with all resource categories. As this resulted in less than 2 months of plant feed, only two further runs were generated to examine the effect of dilution and limiting the resource category to exclude Inferred.

All runs for the North Zone are summarised in Table 5-6 with the Base case (SC303) highlighted.

Table 5-6 South and North Zone Pit Optimisation Runs Focusing on constraints

Scenario	Description	Material						Financials						Mine Life	Strip Ratio	% Inferred
		Ore Mined		Waste	Total	Ounces	Rec Ounces	Undiscounted				Cost per Oz				
		TOTAL ORE						Mining Cost	Processing Cost	Selling Costs	Revenue		Cashflow			
		TOTAL Mt	Au g/t	Mt	Mt	koz	koz					\$M				
South zone																
<a href="#">SCN 201</a>	Undiluted & unconstrained - MIF, 3000 AUD/oz	4.0	1.16	27.9	31.9	147	132	-\$133.6	-\$122.8	\$0.0	\$385.0	\$128.7	-\$1,942	0.0	7.0	2%
<a href="#">SCN 202</a>	Undiluted - within Tenements - MIF, 3000 AUD/oz	0.9	1.19	4.3	5.2	35	32	-\$21.9	-\$28.0	\$0.0	\$93.7	\$43.8	-\$1,555	0.0	4.7	3%
<a href="#">SCN 203</a>	Dilution 0.7m skin - unconstrained - MIF, 3000 AUD/oz	3.8	1.03	23.2	27.0	127	113	-\$112.7	-\$118.7	\$0.0	\$329.6	\$98.1	-\$2,049	1.0	6.0	2%
<a href="#">SCN 204</a>	Dilution 0.7m skin - in Mining tenements - MIF, 3000 AUD/oz	1.0	1.03	4.0	5.1	33	30	-\$21.1	-\$30.5	\$0.0	\$87.9	\$36.3	-\$1,713	0.3	4.0	4%
<a href="#">SCN 205</a>	Dilution 0.7m skin - in Mining Tenements - M&I, 3000 AUD/oz	1.0	0.95	3.6	4.6	30	27	-\$19.0	-\$29.2	\$0.0	\$77.7	\$29.5	-\$1,811	0.2	3.8	0%
<a href="#">SCN 206</a>	Dilution 0.7m skin - Woodland limits - M&I Only, 3000 AUD/oz	2.4	1.02	17.7	20.1	77	69	-\$84.0	-\$72.5	\$0.0	\$202.2	\$45.8	-\$2,257	0.6	7.5	0%
North Zone																
<a href="#">SCN 301</a>	Undiluted - MIF, 3000 AUD/oz	0.4	1.83	5.9	6.3	26	23	-\$27.5	-\$14.6	\$0.0	\$67.2	\$25.0	-\$1,830	0.0	13.2	26%
<a href="#">SCN 302</a>	Dilution 0.7m skin - MIF, 3000 AUD/oz	0.4	1.46	3.7	4.2	19	17	-\$18.3	-\$13.6	\$0.0	\$50.2	\$18.3	-\$1,854	0.0	9.1	24%
<a href="#">SCN 303</a>	Dilution 0.7m skin - M&I Only, 3000 AUD/oz	0.3	1.50	2.6	3.0	15	13	-\$13.0	-\$10.2	\$0.0	\$38.2	\$15.0	-\$1,769	0.0	8.6	0%

#### 5.4.4 Shell Selection

The strategy for the project is to maximise value for 3.6 Mtpa processing plant with a mine life of at least 9 years, which requires a shell with at least 32.4 Mt of ore.

Ausgold selected a A\$3000/oz gold price based on long term consensus forecasting; discounted by approximately 25% to the November 2024 average spot price of ~\$4,100/oz. At the time of reporting in March 2025, the gold price had risen to more than \$4,650/oz.

In order to define economic shells that meet the objectives of the client, larger shells that revenue factor 1 were considered. This was deemed reasonable for the following reasons:

- The objectives outlined in the report “A Guide to Leading Practice in Sustainable Development in Mining,” (DISR, 2011), the approach to selecting the optimisation shell was rooted in sustainable mining practices. By opting for a higher gold price, the aim was to ensure that mining operations were economically viable while also prioritizing environmental and social responsibility. This strategy allowed for maximizing resource recovery in a manner that supported long-term sustainability.
- The base gold price was significantly lower than spot prices and forecasts for gold reaching US\$3000/oz by year end (Sepanek, 2025) with AUD/USD exchange rates expected to decline to 0.63 by year end (Kharitonov, 2025), which equate to an Australian gold price of \$4760/oz.
- Cut-off grade is influenced by the process costs, recoveries and gold prices. A practical diluted cut-off grade of 0.4 g/t was derived to account for these variables so that operations remained flexible and responsive to market conditions, optimising the balance between ore and waste and enhancing the project’s financial performance. At the base price of \$3000/oz, the 0.4 g/t cut-off is no more than 0.04 g/t lower and up to 0.07 g/t higher than the economic cut-off derived using the tails based recovery formula and weathering dependent processing costs.
- One of the challenges faced is the potential sterilisation of ore within future cutbacks that are too narrow to mine practically. By selecting an optimisation shell that accounts for a slightly higher gold price, the design of pit phases minimizes the sterilisation of valuable ore. This approach allows for more effectively planning of future expansions and cutbacks, ensuring that significant quantities of ore that could otherwise be extracted are not left behind.

The selection of the open pit optimisation shell based on a higher gold price is a strategic decision aimed at maximising resource extraction, maintaining operational efficiency, and planning for future mining activities. The shells selected for design are summarised in Table 5-7.

Table 5-7 Summary of Selected Shell Inventories

Scenario	Model	Optimisation Shell	Revenue Factor	Material					
				Ore Mined		Waste	Total	Ounces	Rec Ounces
				TOTAL ORE					
TOTAL Mt	Au g/t	Mt	Mt	koz	koz				
SCN 103	CZ	44	1.14	29.4	1.15	195.7	225.0	1,081	974
SCN 206	SZ	43	1.14	2.7	1.02	22.1	24.8	89	79
SCN 303	NZ	21	0.98	0.3	1.52	2.1	2.3	13	12
<b>TOTAL</b>				<b>32.3</b>	<b>1.14</b>	<b>219.8</b>	<b>252.2</b>	<b>1,183</b>	<b>1,065</b>

CZ is the largest pit and provides more than 90% of the total material mined, ore and ounces for the project. CZ optimisation is therefore the focus for increasing the mine life by selecting a larger shell. Scenario 103 as highlighted in Table 5-8 and Table 5-9 provides the basis for shell selection. The revenue factor 1.14 or Shell 43 (highlighted in pink) was selected as shell meets the criteria for a provides for the following:

- Adds 3.4 Mt at 0.84 g/t which extends the mine life of CZ from 6.5 years to 7.3 years.
- Adds 80 koz recovered at a cost of \$3,122/oz (4% higher than the base price).
- Minimises potential losses from future cutbacks as shown in Figure 5-9 where the expanded outline for \$3420/oz shell (RF 1.14) is very close to the \$3000/oz shell with adequate room in most places for potential expansion to the \$5000/oz shell.

The revenue factor 1 shell was selected for NZ as this pit does not have sufficient ore to make a meaningful difference to the mine life.

SZ supplies the remaining feed for the mine life. Scenario 206 as highlighted in Table 5-9 and Figure 5-10 provides the basis for shell selection The revenue factor of 1.14 or shell 42 (highlighted) was selected for design in line with the CZ shell selection, and adds:

- 0.3 Mt at 1.0 g/t which extends the mine life by approximately one month.
- 10 koz recovered at a cost of \$3167/oz (less than 6% higher than the base price).
- Minimises potential losses from future cutbacks as shown in Figure 5-11 where the expanded outline for \$3420/oz shell (RF 1.14) is very close to the \$3000/oz shell.

Table 5-8: Scenario 103 Central Zone Results (showing highest, lowest and average discounted cashflows with selected shell highlighted in pink)

Shell	Revenue Factor	ORE		Mined Ounces	Rec Ounces	Waste	Total	Financials (Undiscounted)				Discounted Cashflows			Selection Criteria	
		Ore Mt	Au g/t					kOz	kOz	Mt	Mt	Mining Cost (\$M)	Ore Process Cost (\$M)	Revenue (\$M)	Cashflow (\$M)	Best Case (\$M)
				Overall	Overall											
1	0.3	2.4	1.95	153	142	8.3	10.7	-\$46.7	-\$74.6	\$413.9	\$292.6	\$274.3	\$274.3	\$274.3	120.2	3.41
2	0.32	2.6	1.94	161	150	8.9	11.4	-\$50.0	-\$79.3	\$437.0	\$307.7	\$287.3	\$287.3	\$287.3	118.9	3.42
3	0.34	2.9	1.87	175	162	9.5	12.4	-\$54.3	-\$89.2	\$473.5	\$330.1	\$305.6	\$305.6	\$305.6	113.5	3.26
4	0.36	3.3	1.85	197	182	11.4	14.7	-\$64.6	-\$101.3	\$532.0	\$366.2	\$335.5	\$335.5	\$335.5	110.9	3.44
5	0.38	3.6	1.80	210	194	12.1	15.7	-\$69.4	-\$111.1	\$566.5	\$386.0	\$350.9	\$350.9	\$350.9	106.6	3.35
6	0.4	3.9	1.76	220	204	12.7	16.6	-\$73.1	-\$119.5	\$594.1	\$401.5	\$364.9	\$364.6	\$364.7	102.9	3.25
7	0.42	4.0	1.74	224	207	12.8	16.8	-\$74.4	-\$122.3	\$603.2	\$406.4	\$369.3	\$368.8	\$369.0	101.7	3.21
8	0.44	4.5	1.68	243	225	14.4	18.9	-\$83.7	-\$137.8	\$655.3	\$433.8	\$393.3	\$390.9	\$392.1	96.4	3.20
9	0.46	4.8	1.69	262	242	17.2	22.0	-\$97.6	-\$147.3	\$705.3	\$460.4	\$416.3	\$412.7	\$414.5	95.6	3.58
10	0.48	5.0	1.67	268	247	17.7	22.7	-\$100.7	-\$152.5	\$721.5	\$468.4	\$423.1	\$418.6	\$420.9	94.0	3.56
11	0.5	5.2	1.65	276	255	18.4	23.7	-\$105.0	-\$159.3	\$742.9	\$478.6	\$431.5	\$425.8	\$428.6	91.9	3.54
12	0.52	5.5	1.63	288	266	20.1	25.5	-\$113.6	-\$167.9	\$774.9	\$493.4	\$443.7	\$436.5	\$440.1	89.9	3.66
13	0.54	5.9	1.60	303	278	21.7	27.6	-\$122.7	-\$180.0	\$812.1	\$509.5	\$456.5	\$447.1	\$451.8	86.7	3.69
14	0.56	6.0	1.59	307	282	22.1	28.1	-\$125.2	-\$184.2	\$823.8	\$514.5	\$460.3	\$450.0	\$455.1	85.5	3.67
15	0.58	6.5	1.55	325	299	24.4	30.9	-\$138.0	-\$199.8	\$871.4	\$533.6	\$475.0	\$461.4	\$468.2	81.9	3.74
16	0.6	6.8	1.53	336	308	25.8	32.6	-\$145.8	-\$208.6	\$898.5	\$544.1	\$482.8	\$467.3	\$475.0	80.0	3.79
17	0.62	9.4	1.49	451	413	47.4	56.8	-\$255.0	-\$288.8	\$1,204.5	\$660.8	\$570.5	\$527.7	\$549.1	70.1	5.03
18	0.64	9.7	1.47	460	421	48.4	58.1	-\$260.7	-\$298.1	\$1,227.6	\$668.8	\$575.7	\$530.6	\$553.2	68.8	4.97
19	0.66	10.1	1.45	474	433	50.5	60.6	-\$272.4	-\$310.9	\$1,264.5	\$681.2	\$583.9	\$534.3	\$559.1	67.2	4.98
20	0.68	12.4	1.45	578	529	74.3	86.8	-\$394.8	-\$381.7	\$1,543.9	\$767.4	\$642.0	\$563.8	\$602.9	61.7	5.97
21	0.7	12.8	1.43	590	539	76.2	89.0	-\$404.9	-\$392.1	\$1,573.0	\$776.0	\$647.4	\$565.5	\$606.4	60.7	5.96
22	0.72	13.1	1.42	598	547	77.3	90.4	-\$411.3	-\$402.1	\$1,595.9	\$782.4	\$651.0	\$566.7	\$608.9	59.6	5.89
23	0.74	13.7	1.40	617	563	80.5	94.2	-\$429.4	-\$419.6	\$1,644.0	\$795.0	\$658.3	\$570.7	\$614.5	58.1	5.88
WORST CASE - 24	0.76	14.0	1.39	624	570	81.6	95.6	-\$435.8	-\$427.6	\$1,663.0	\$799.6	\$660.7	\$571.9	\$616.3	57.3	5.85
25	0.78	16.9	1.31	713	650	99.0	116.0	-\$527.1	-\$518.3	\$1,896.6	\$851.2	\$691.6	\$560.6	\$626.1	50.3	5.85
26	0.8	17.3	1.30	725	660	101.2	118.5	-\$538.6	-\$530.3	\$1,926.1	\$857.2	\$695.0	\$560.0	\$627.5	49.5	5.84
AVERAGE CASE - 27	0.82	17.8	1.29	738	672	103.8	121.6	-\$552.7	-\$544.7	\$1,961.2	\$863.8	\$698.4	\$558.8	\$628.6	48.5	5.83
28	0.84	18.8	1.28	773	702	112.6	131.3	-\$596.5	-\$574.7	\$2,049.5	\$878.3	\$707.0	\$548.8	\$627.9	46.8	6.00
29	0.86	19.2	1.27	784	713	114.7	133.9	-\$608.3	-\$587.9	\$2,078.9	\$882.7	\$709.4	\$545.4	\$627.4	46.0	5.97
30	0.88	20.8	1.24	830	753	125.8	146.5	-\$664.3	-\$635.5	\$2,198.1	\$898.4	\$717.8	\$526.0	\$621.9	43.3	6.06
31	0.9	21.5	1.23	850	771	129.7	151.3	-\$686.5	-\$659.1	\$2,249.8	\$904.1	\$720.5	\$520.9	\$620.7	42.0	6.03
32	0.92	23.8	1.20	920	834	147.7	171.4	-\$782.9	-\$728.5	\$2,432.4	\$921.1	\$729.4	\$499.0	\$614.2	38.7	6.21
33	0.94	24.1	1.20	929	842	150.0	174.1	-\$795.6	-\$739.3	\$2,457.8	\$922.9	\$730.3	\$495.9	\$613.1	38.2	6.21
34	0.96	24.8	1.19	949	860	155.2	180.0	-\$822.7	-\$760.9	\$2,509.2	\$925.7	\$731.5	\$485.9	\$608.7	37.3	6.25
35	0.98	25.1	1.19	955	865	156.3	181.3	-\$829.1	-\$767.9	\$2,523.1	\$926.1	\$731.7	\$483.4	\$607.5	37.0	6.24
BEST CASE - 36	1	26.0	1.19	990	895	167.8	193.7	-\$886.4	-\$796.9	\$2,610.2	\$926.8	\$732.1	\$464.2	\$598.1	35.7	6.46
37	1.02	26.6	1.18	1,008	910	172.8	199.4	-\$912.1	-\$817.7	\$2,656.1	\$926.3	\$731.8	\$451.6	\$591.7	34.8	6.48
38	1.04	26.9	1.17	1,015	916	174.9	201.8	-\$923.1	-\$824.6	\$2,673.5	\$925.8	\$731.5	\$447.1	\$589.3	34.4	6.51
39	1.06	27.3	1.17	1,027	927	178.7	206.0	-\$942.3	-\$838.0	\$2,704.5	\$924.2	\$730.8	\$437.1	\$583.9	33.8	6.54
40	1.08	27.7	1.16	1,038	937	182.0	209.8	-\$959.9	-\$851.4	\$2,733.3	\$922.0	\$729.8	\$428.3	\$579.0	33.2	6.56
41	1.1	28.4	1.16	1,055	951	187.0	215.4	-\$985.7	-\$871.4	\$2,775.2	\$918.0	\$727.9	\$415.0	\$571.5	32.3	6.59
42	1.12	29.0	1.15	1,073	967	193.0	222.0	-\$1,016.8	-\$891.2	\$2,821.3	\$913.3	\$725.7	\$402.6	\$564.2	31.5	6.65
Selected Shell - 43	1.14	29.4	1.15	1,081	974	195.7	225.0	-\$1,031.0	-\$901.4	\$2,842.9	\$910.4	\$724.5	\$396.0	\$560.2	31.0	6.67
44	1.16	30.5	1.13	1,112	1,001	205.8	236.3	-\$1,085.9	-\$937.5	\$2,921.6	\$898.3	\$719.3	\$371.6	\$545.5	29.4	6.74
45	1.18	31.6	1.12	1,140	1,026	214.7	246.4	-\$1,135.0	-\$971.7	\$2,992.5	\$885.8	\$714.1	\$354.5	\$534.3	28.0	6.79
46	1.2	31.9	1.12	1,148	1,033	218.0	249.9	-\$1,151.7	-\$979.9	\$3,013.4	\$881.8	\$712.5	\$347.6	\$530.0	27.6	6.83
47	1.22	32.3	1.12	1,158	1,041	221.6	253.8	-\$1,170.0	-\$991.4	\$3,038.0	\$876.5	\$710.3	\$338.3	\$524.3	27.2	6.87
48	1.24	32.7	1.11	1,167	1,049	224.8	257.5	-\$1,186.6	-\$1,003.4	\$3,061.2	\$871.1	\$708.2	\$328.2	\$518.2	26.7	6.89
49	1.26	33.6	1.10	1,190	1,068	232.9	266.5	-\$1,227.9	-\$1,032.4	\$3,117.1	\$856.7	\$702.7	\$305.3	\$504.0	25.5	6.93
50	1.28	33.7	1.10	1,193	1,071	234.1	267.8	-\$1,234.1	-\$1,037.0	\$3,125.5	\$854.3	\$701.8	\$302.0	\$501.9	25.3	6.94

Table 5-9 Scenario 206 South Zone Results (showing highest / lowest discounted cashflows with selected shell highlighted in pink)

Shell	Revenue Factor	ORE		Mined Ounces kOz	Rec Ounces kOz	Waste Mt	Total Mt	Financials (Undiscounted)				Discounted Cashflows			Selection Criteria	
		Ore Mt	Au g/t					Mining Cost (\$M)	Ore Process Cost (\$M)	Revenue (\$M)	Cashflow (\$M)	Best Case (\$M)	Worst Case (\$M)	Average Case (\$M)	Undiscounted Profit / Tonne Ore	Strip Ratio
															Overall	Overall
1	0.3	0.1	1.59	5	5	0.1	0.2	-\$0.8	-\$2.9	\$13.7	\$10.0	\$10.0	\$10.0	\$10.0	101.0	0.82
2	0.32	0.1	1.48	7	6	0.1	0.3	-\$1.1	-\$4.2	\$18.3	\$13.0	\$13.0	\$13.0	\$13.0	91.0	0.84
3	0.34	0.2	1.44	7	7	0.1	0.3	-\$1.3	-\$4.7	\$20.1	\$14.2	\$14.2	\$14.2	\$14.2	87.9	0.88
4	0.36	0.2	1.41	8	7	0.1	0.3	-\$1.4	-\$5.1	\$21.4	\$15.0	\$15.0	\$15.0	\$15.0	85.2	0.83
5	0.38	0.2	1.41	8	7	0.2	0.3	-\$1.4	-\$5.2	\$21.6	\$15.1	\$15.1	\$15.1	\$15.1	85.1	0.85
6	0.4	0.2	1.40	8	8	0.2	0.3	-\$1.5	-\$5.3	\$22.1	\$15.3	\$15.3	\$15.3	\$15.3	84.0	0.85
7	0.42	0.2	1.40	8	8	0.2	0.3	-\$1.5	-\$5.4	\$22.3	\$15.4	\$15.4	\$15.4	\$15.4	83.8	0.87
8	0.44	0.2	1.37	9	8	0.2	0.4	-\$1.6	-\$5.8	\$23.5	\$16.1	\$16.1	\$16.1	\$16.1	81.0	0.92
9	0.46	0.2	1.35	9	9	0.2	0.4	-\$1.8	-\$6.2	\$24.9	\$16.8	\$16.8	\$16.8	\$16.8	78.7	1.00
10	0.48	0.2	1.34	9	9	0.2	0.4	-\$1.9	-\$6.4	\$25.4	\$17.1	\$17.1	\$17.1	\$17.1	77.7	1.02
11	0.5	0.3	1.27	11	10	0.3	0.6	-\$2.6	-\$7.9	\$29.5	\$19.0	\$19.0	\$19.0	\$19.0	70.8	1.26
12	0.52	0.3	1.25	11	11	0.4	0.7	-\$2.8	-\$8.4	\$30.8	\$19.6	\$19.6	\$19.6	\$19.6	68.5	1.28
13	0.54	0.3	1.24	12	11	0.4	0.7	-\$2.9	-\$8.7	\$31.5	\$19.9	\$19.9	\$19.9	\$19.9	67.4	1.31
14	0.56	0.3	1.23	12	11	0.4	0.7	-\$2.9	-\$8.8	\$31.8	\$20.0	\$20.0	\$20.0	\$20.0	66.8	1.30
15	0.58	0.3	1.23	12	11	0.4	0.7	-\$3.0	-\$8.8	\$31.8	\$20.1	\$20.1	\$20.1	\$20.1	66.7	1.30
16	0.6	0.4	1.17	14	13	0.6	1.0	-\$4.4	-\$11.3	\$38.4	\$22.7	\$22.7	\$22.7	\$22.7	59.1	1.69
17	0.62	0.4	1.16	15	13	0.7	1.1	-\$4.5	-\$11.5	\$38.9	\$22.8	\$22.8	\$22.8	\$22.8	58.8	1.74
18	0.64	0.4	1.13	16	14	0.8	1.2	-\$5.2	-\$12.6	\$41.6	\$23.8	\$23.8	\$23.8	\$23.8	55.7	1.82
19	0.66	0.5	1.11	17	15	0.9	1.4	-\$6.1	-\$13.7	\$44.7	\$24.9	\$24.9	\$24.9	\$24.9	53.2	2.01
20	0.68	0.5	1.10	18	16	1.1	1.6	-\$7.0	-\$14.8	\$47.6	\$25.9	\$25.9	\$25.9	\$25.9	51.3	2.22
21	0.7	0.5	1.10	18	16	1.1	1.6	-\$7.0	-\$14.8	\$47.7	\$25.9	\$25.9	\$25.9	\$25.9	51.3	2.23
22	0.72	0.5	1.10	18	17	1.1	1.7	-\$7.2	-\$15.0	\$48.1	\$26.0	\$26.0	\$26.0	\$26.0	51.0	2.25
23	0.74	0.5	1.08	19	17	1.2	1.8	-\$7.6	-\$15.8	\$49.9	\$26.5	\$26.5	\$26.5	\$26.5	49.0	2.27
24	0.76	0.5	1.08	19	17	1.2	1.8	-\$7.7	-\$15.9	\$50.2	\$26.5	\$26.5	\$26.5	\$26.5	48.9	2.29
25	0.78	0.6	1.07	19	18	1.3	1.9	-\$8.2	-\$16.5	\$51.6	\$26.8	\$26.8	\$26.8	\$26.8	47.7	2.39
26	0.8	0.6	1.06	20	18	1.4	2.0	-\$8.6	-\$16.9	\$52.5	\$27.0	\$27.0	\$27.0	\$27.0	47.0	2.44
27	0.82	0.6	1.06	20	18	1.4	2.0	-\$8.6	-\$17.0	\$52.7	\$27.1	\$27.1	\$27.1	\$27.1	46.7	2.45
28	0.84	0.6	1.06	20	18	1.5	2.1	-\$9.0	-\$17.3	\$53.5	\$27.2	\$27.2	\$27.2	\$27.2	46.1	2.51
29	0.86	0.6	1.05	20	18	1.5	2.1	-\$9.1	-\$17.5	\$53.8	\$27.3	\$27.3	\$27.3	\$27.3	45.6	2.51
30	0.88	2.2	1.04	74	66	16.8	19.0	-\$79.1	-\$67.6	\$192.0	\$45.3	\$45.3	\$45.3	\$45.3	20.5	7.62
31	0.9	2.2	1.03	74	66	16.9	19.1	-\$79.6	-\$68.3	\$193.3	\$45.4	\$45.4	\$45.4	\$45.4	20.4	7.58
32	0.92	2.3	1.03	75	67	17.0	19.3	-\$80.2	-\$69.2	\$194.9	\$45.5	\$45.5	\$45.5	\$45.5	20.2	7.53
33	0.94	2.3	1.02	76	68	17.2	19.5	-\$81.3	-\$70.3	\$197.3	\$45.7	\$45.7	\$45.7	\$45.7	19.9	7.50
34	0.96	2.3	1.02	76	68	17.3	19.6	-\$81.5	-\$70.5	\$197.7	\$45.7	\$45.7	\$45.7	\$45.7	19.8	7.50
35	0.98	2.3	1.02	76	68	17.4	19.7	-\$82.3	-\$71.3	\$199.3	\$45.8	\$45.8	\$45.8	\$45.8	19.6	7.47
<b>BEST/WORST/AVER - 36</b>	<b>1.02</b>	<b>2.4</b>	<b>1.02</b>	<b>77</b>	<b>69</b>	<b>17.7</b>	<b>20.1</b>	<b>-\$84.0</b>	<b>-\$72.5</b>	<b>\$202.2</b>	<b>\$45.8</b>	<b>\$45.8</b>	<b>\$45.8</b>	<b>\$45.8</b>	<b>19.3</b>	<b>7.48</b>
37	1.04	2.4	1.02	80	71	18.5	20.9	-\$87.6	-\$74.3	\$207.5	\$45.7	\$45.7	\$45.7	\$45.7	18.8	7.61
38	1.06	2.4	1.02	80	71	18.6	21.0	-\$88.1	-\$74.7	\$208.4	\$45.6	\$45.6	\$45.6	\$45.6	18.7	7.60
39	1.08	2.5	1.01	81	72	18.9	21.4	-\$89.7	-\$75.9	\$211.0	\$45.4	\$45.4	\$45.4	\$45.4	18.3	7.61
40	1.1	2.6	1.01	83	75	20.0	22.6	-\$94.5	-\$78.3	\$217.7	\$44.8	\$44.8	\$44.8	\$44.8	17.5	7.81
41	1.12	2.7	1.02	88	79	21.9	24.6	-\$103.3	-\$82.3	\$229.2	\$43.6	\$43.6	\$43.6	\$43.6	16.2	8.15
<b>Selected Shell - 42</b>	<b>1.14</b>	<b>2.7</b>	<b>1.02</b>	<b>89</b>	<b>79</b>	<b>22.1</b>	<b>24.8</b>	<b>-\$104.3</b>	<b>-\$83.1</b>	<b>\$230.8</b>	<b>\$43.4</b>	<b>\$43.4</b>	<b>\$43.4</b>	<b>\$43.4</b>	<b>16.0</b>	<b>8.15</b>
43	1.16	2.7	1.01	89	79	22.2	24.9	-\$104.6	-\$83.4	\$231.3	\$43.3	\$43.3	\$43.3	\$43.3	15.9	8.14
44	1.18	2.7	1.01	89	79	22.2	25.0	-\$104.9	-\$83.6	\$231.7	\$43.2	\$43.2	\$43.2	\$43.2	15.8	8.15
45	1.2	2.8	1.01	91	82	23.4	26.2	-\$110.0	-\$86.0	\$238.0	\$42.0	\$42.0	\$42.0	\$42.0	15.0	8.33
46	1.22	2.8	1.01	92	82	23.5	26.3	-\$110.5	-\$86.3	\$238.7	\$41.9	\$41.9	\$41.9	\$41.9	14.9	8.34
47	1.24	2.9	1.00	95	85	24.6	27.6	-\$116.4	-\$90.2	\$246.6	\$40.0	\$40.0	\$40.0	\$40.0	13.6	8.37
48	1.26	3.0	1.00	95	85	24.7	27.7	-\$116.8	-\$90.5	\$247.2	\$39.8	\$39.8	\$39.8	\$39.8	13.5	8.38
49	1.28	3.0	1.00	95	85	24.8	27.7	-\$116.9	-\$90.5	\$247.3	\$39.8	\$39.8	\$39.8	\$39.8	13.5	8.38
50	1.3	3.0	1.00	95	85	24.8	27.7	-\$117.1	-\$90.6	\$247.5	\$39.8	\$39.8	\$39.8	\$39.8	13.5	8.39

Figure 5-8 Scenario 103 Central Zone Base Case Optimisation

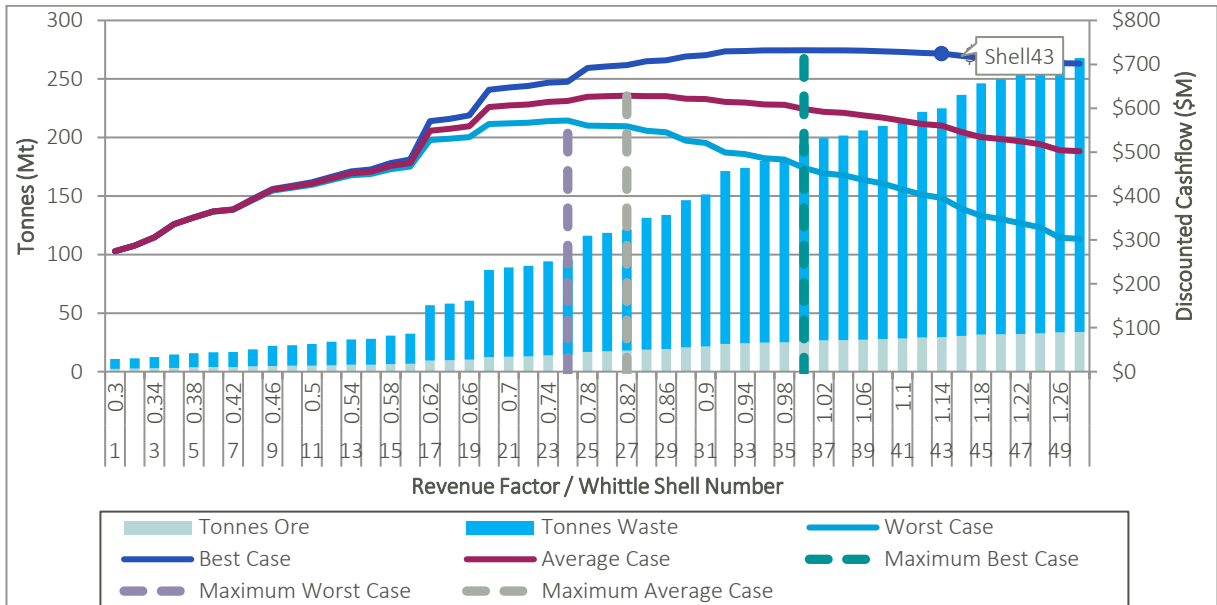


Figure 5-9 Comparison of Base case and Selected shells for CZ

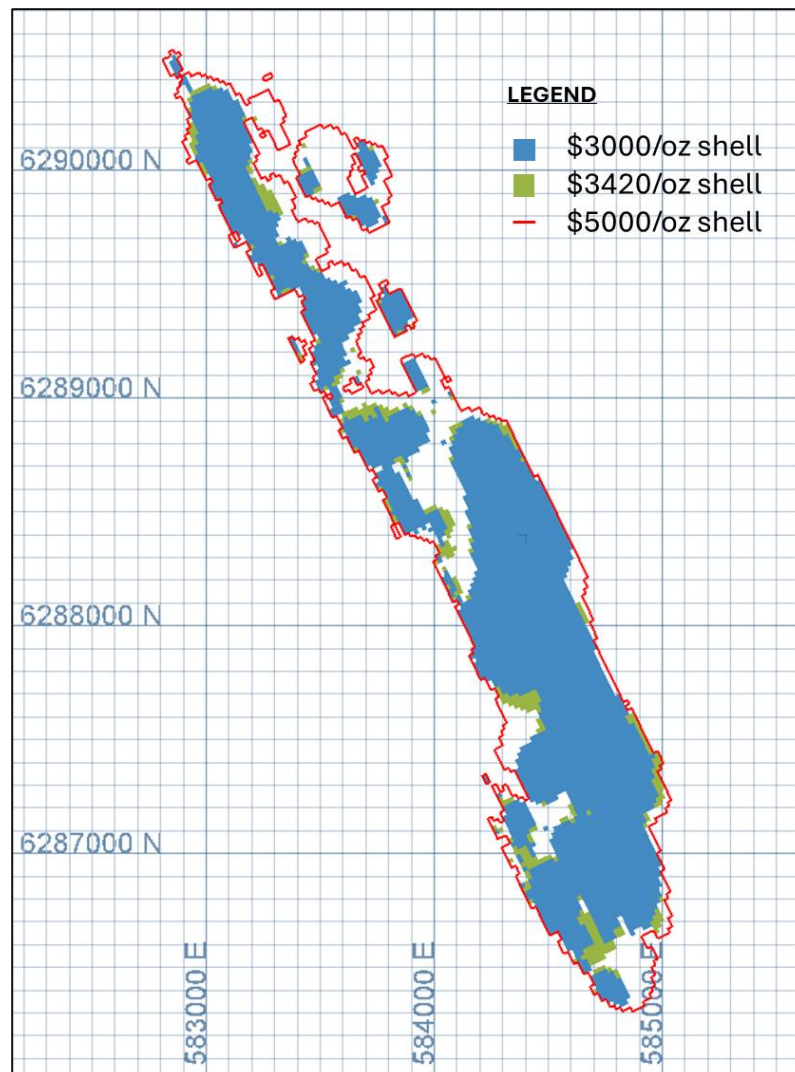


Figure 5-10 Scenario 206 South Zone Base Case Optimisation

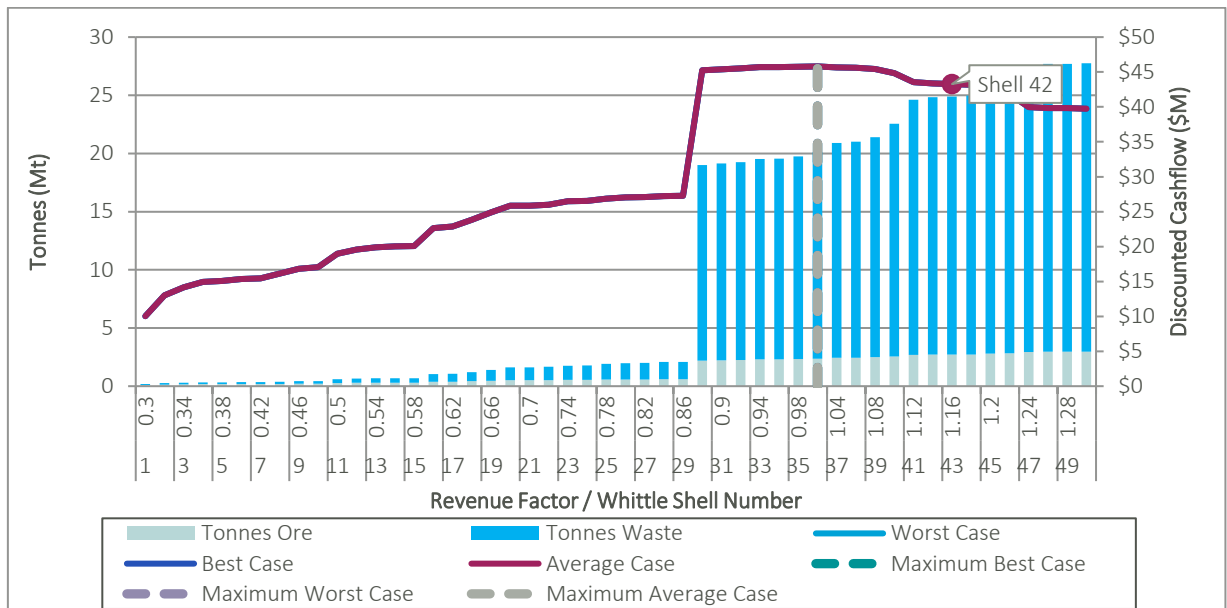
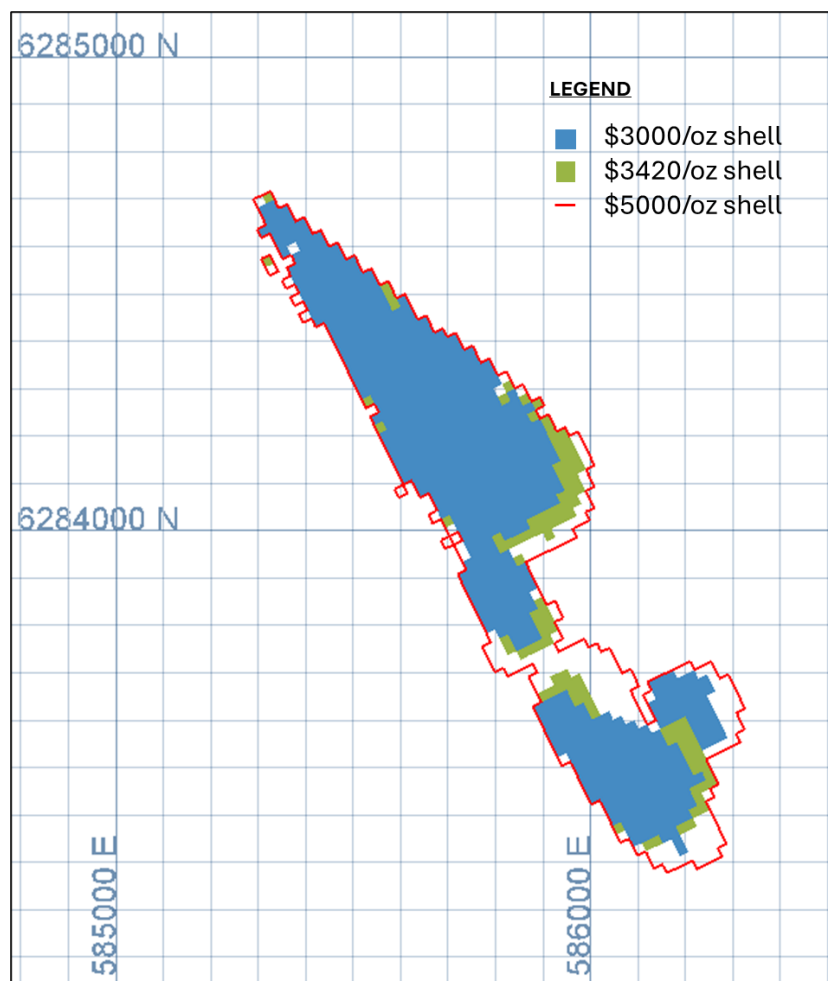


Figure 5-11 Comparison of Base case and Selected shells for SZ



## 6 MINE DESIGN

### 6.1 Mine Design Criteria

#### 6.1.1 Design Objective

The design aims to optimize resource extraction while ensuring safety, efficiency, and environmental sustainability. This following sections outlines the key aspects of the open pit mine design criteria and physicals, pit inventories, waste dump design, supporting infrastructure and general site layout.

#### 6.1.2 Open Pit Slopes

The 90° batter slopes recommended by 3<sup>rd</sup> Rock Consulting (Section 3.4.4) are impractical for design purposes, therefore, for the DFS pit designs, the 90° batter slopes were adjusted to 80° with a subsequent reduction in the berm width to 5.0 m to maintain inter-ramp slope angle. In practice, this berm is too narrow to effectively catch and contain small scale failures, and a steeper batter slope and wider berm would need to be considered in operations. The designs are not for construction and should be reviewed prior to implementation. The revised slope profiles used for design are summarised in Table 6-1.

Table 6-1 Revised Open Pit Slope Design Parameters

Deposit	Weathering	Pit	Lithology (slope dip direction)	Batter Height (m)	Berm Width (m)	Bench face angle	Inter-ramp slope angle
All	Saprolite	All	All	10	6.5	55	36.5
	Transitional		All	20	8.5	75	55.3
North	Fresh	Jackson & Olympia	NF1 (225°-295°)	20	8.5	80	59.0
			NF2 (0°-225° & 295°-360°)	20	5.0	80	67.0
		Jinkas Main	CF1 (230°-260°)	20	8.5	80	59.0
		Jinkas South & White Dam	CF2 (0°-230° & 260°-360°)	20	5.0	80	67.0
South	Fresh	Dingo	SF1 (230°-260°)	20	8.5	85	59.0
			SF2 (0°-230° & 260°-360°)	20	5.0	80	67.0

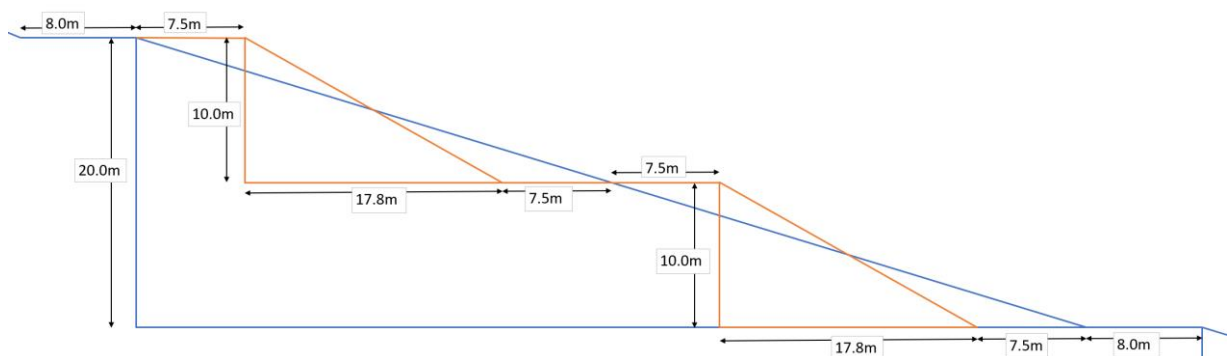
### 6.1.3 Waste dump slopes

The geotechnical recommendations for waste dump rehabilitation were non-specific requiring that final slopes are reprofiled to 20° or less. When considering the regional topography, a flatter overall slope of 16° was targeted with battered final slopes of 18°, as this was deemed more likely to be accepted by regulators. In terms of the waste dump height, this can be up to 80.0 m high. The design parameters for the as-built and final slopes and shown in Table 6-2 and Figure 6-1.

Table 6-2: Waste Dump Design Criteria

Wall Location	Lift Height	Berm Width	Batter Angle	Inter-ramp slope angle	Overall Slope Angle
	m	m	°	°	°
As Built	10	15 / 23	36	16	19.6
Rehabilitated	20	8	18	16	16.5

Figure 6-1 As-built and final rehabilitated slope dimensions



### 6.1.4 Ramps

Ramp widths for operating equipment are based on 140 t trucks as defined in Table 6-3. Surface haul roads are 30 m wide based on an operating width of 25 m with 2.5 m wide shoulder on either side.

Most ramps are dual lane to provide sufficient access to meet productivity targets. Single-lane ramp access is generally used for benches at the base of the pits, limited to 200 m in length, due to lower traffic intensity in these areas.

The ramp design configuration, based on the selected Caterpillar 785D dump truck, are presented in Figure 6-2 and Figure 6-3.

Table 6-3: Designed Ramp Width's

Road Type	Truck		Bund	Drain	Minimum Running Width		Total Tamp Width (TRW)
	Type	Operating Width (TOW)	Overall Width (BW)	Width (DW)	Multiplier	Width (RW) <sup>2</sup>	
		m	m	m	Value	m	M (Rounded)
Dual lane	Caterpillar 785D	7.06	4.5	1.0	3.5	24.7	30.0
Single lane	Caterpillar 785D	7.06	4.5	1	1.5	10.6	16.0

Figure 6-2 Dual Lane Ramp Layout

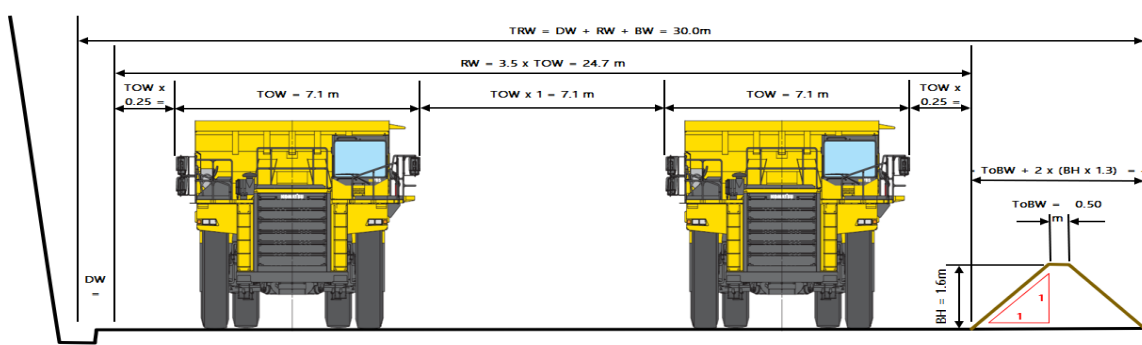
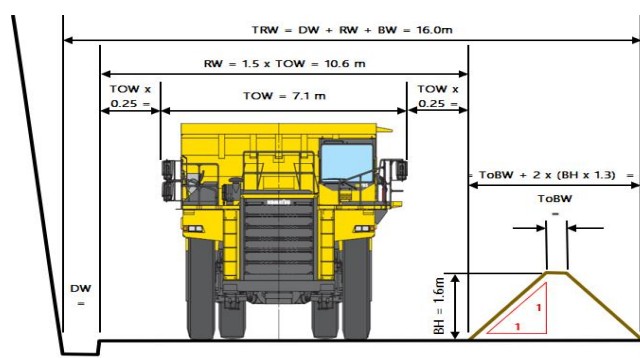


Figure 6-3 Single Lane Ramp Layout



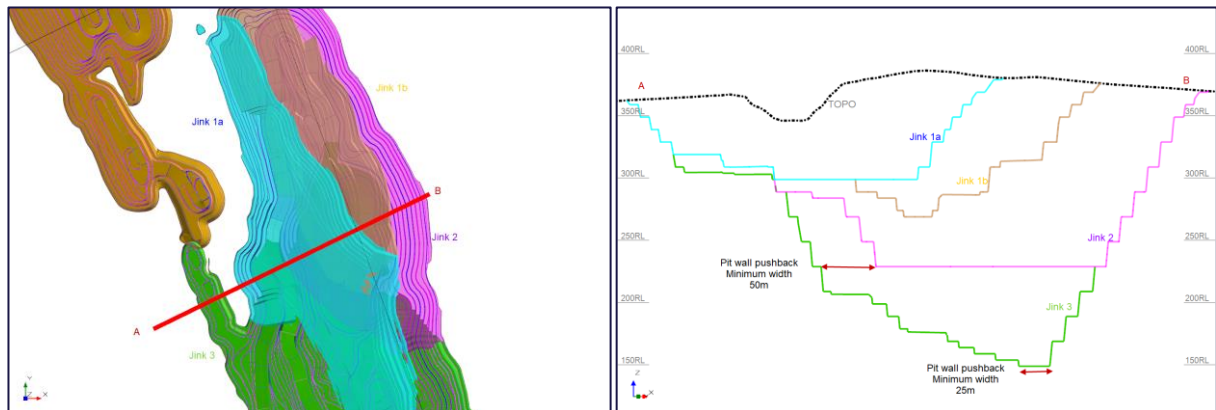
### 6.1.5 Minimum Mining Width

The pit design prioritises safe, efficient, and practical ore and waste extraction. A minimum mining width is maintained at all stages to ensure sufficient space for equipment access and to prevent congestion. Based on a 250 t-class excavator and 140 t-class trucks, the following mining width constraints are applied:

- **Pit wall pushback:** A minimum safe mining width of 80 m is maintained between successive pit wall pushbacks for interim stages. This applies only to the Jinkas mining area, where internal pit stage designs have been developed.
- **Base of pit:** A minimum width of 25 m is applied at the bottom benches to allow for machine clearance and excavator slew. Final bench "goodbye" cuts, up to 5 m deep,

are designed when sufficient length allows for temporary excavator access. These cuts will be retreat-mined before pit abandonment. (see Figure 6-4)

**Figure 6-4 Minimum Mining Width**



## 6.2 Open Pit Mine Design Physicals

### 6.2.1 General

The optimisation shells identified in section 5.4 were used as a guide for the final pit designs. Jinkas, in the Central Zone, is the only deposit where internal stages were designed, targeting high-value areas identified in shells from lower revenue factors. The designs produced adhered to the criteria and mining practicalities outlined in Section 7-1, aiming to create practical pit layouts that maximise ore tonnage. While efforts were made to minimise waste, this was not prioritised over ore recovery.

The following section matches the writing order and describes the Jinkas, Jackson and Olympia pit designs in the Central Zone and the Dingo pits in the South Zone, including their layouts and cross-sections comparing the designs to the Whittle shells.

### 6.2.2 Jinkas Mine Area

Jinkas is the largest pit and the primary value driver for the project. As a result, it has undergone extensive design evaluation and has been divided into six stages to optimize development (Figure 6-5). This staged approach reduces the initial strip ratio and accelerates access to higher-grade material, unlocking value earlier. The stages were designed based on optimization shells at lower revenue factors, prioritizing higher-margin areas as follows:

- Jinkas 1a and 1b target oxide material for early mill feed, along with higher-grade transitional and fresh material.

- Jinkas 2 is a pushback in the northeast section of the pit.
- Jinkas 3 is the deepest stage, reaching approximately 220 m in depth.
- Jinkas 4 focuses on extracting higher-grade material from the southern part of the pit.
- Jinkas 5 completes the development of the ultimate pit and extends mining to the southern pod.
- Jinkas 6 is the smallest and lowest-value stage, located at the southernmost end of the pit.

Relatively good outcomes were obtained for the design of the Jinkas pits. In these types of pits, which feature complex geometry and multiple stages requiring additional independent ramp access, a waste increase of up to 15% beyond the shell is considered a favourable result. The Jinkas designs captured an extra 68,300 oz from 8.7% more ore at an average grade of 0.98 g/t at the cost of 12% additional waste resulting in an increase in strip ratio from 6.6:1 to 6.8:1. The shell comparison results are presented in Table 6-4.

**Table 6-4 Jinkas Shell to Design Comparison**

Design vs Shell	Material	Grand Total			
		Volume (Mbcm)	Mass (Mt)	DAU (g/t)	Au (koz)
Jinkas_Design	Total Ore	9.6	27.1	1.1	980.6
	WASTE	71.8	183.3		
	Total Rock	81.4	210.4		
Jinkas_SCN103_S43	Total Ore	8.8	25.0	1.1	912.3
	WASTE	63.9	163.5		
	Total Rock	72.7	188.5		
Difference (%)	Total Ore	<b>8.8%</b>	<b>8.7%</b>	<b>-1.1%</b>	<b>7.5%</b>
	Waste	<b>12.4%</b>	<b>12.1%</b>	<b>0.0%</b>	

To achieve an optimal shell-to-design conversion factor with the tight distance between the shell and tenement boundaries on the north-eastern side of the pit, the permanent ramps in that section were placed on the flatter slope on western footwall. This necessitated retreat mining of a section of the Jinkas 2 ramp, where the temporary ramp placed on the hanging wall of Jinkas 1b internal stage is subsequently retreat-mined for access to the material left behind on the hanging wall of Jinkas 2. This approach allowed a deeper and more economic ultimate pit design in the northern section of the ultimate pit by avoiding placement of a permanent ramp on its hanging wall. As a result of this design approach, a scheduling constraint was imposed where a maximum bench lag of 80 m had to be maintained between Jinkas 1b and Jinkas 2 stages to avoid an uneconomic haulage downwards and then upwards when stripping the majority waste material left on the upper benches of Jinkas 2.

Figure 6-6 presents a detailed comparison of the plan view and selected cross-sections, highlighting the optimal shell SCN 103, Shell 43, which formed the basis of the design.

Figure 6-5 Pit Design – Jinkas Mine Area

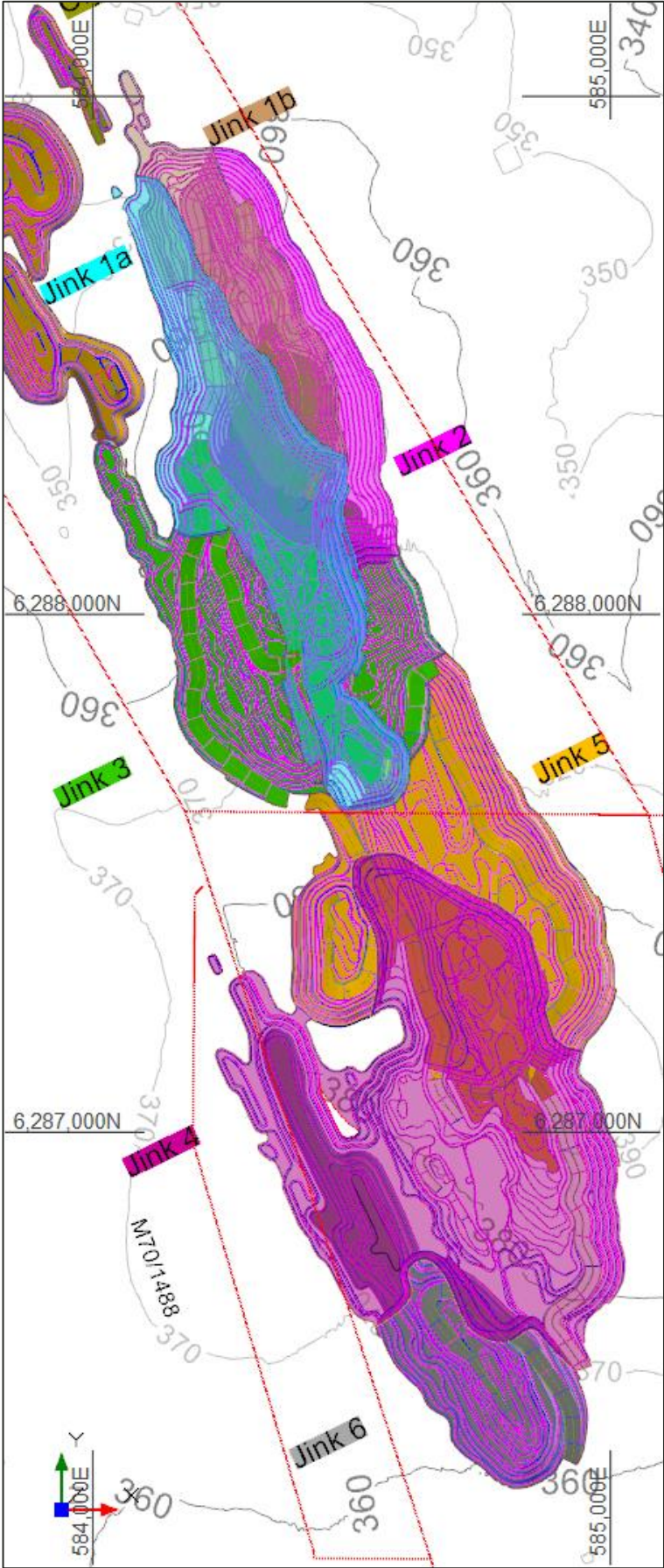
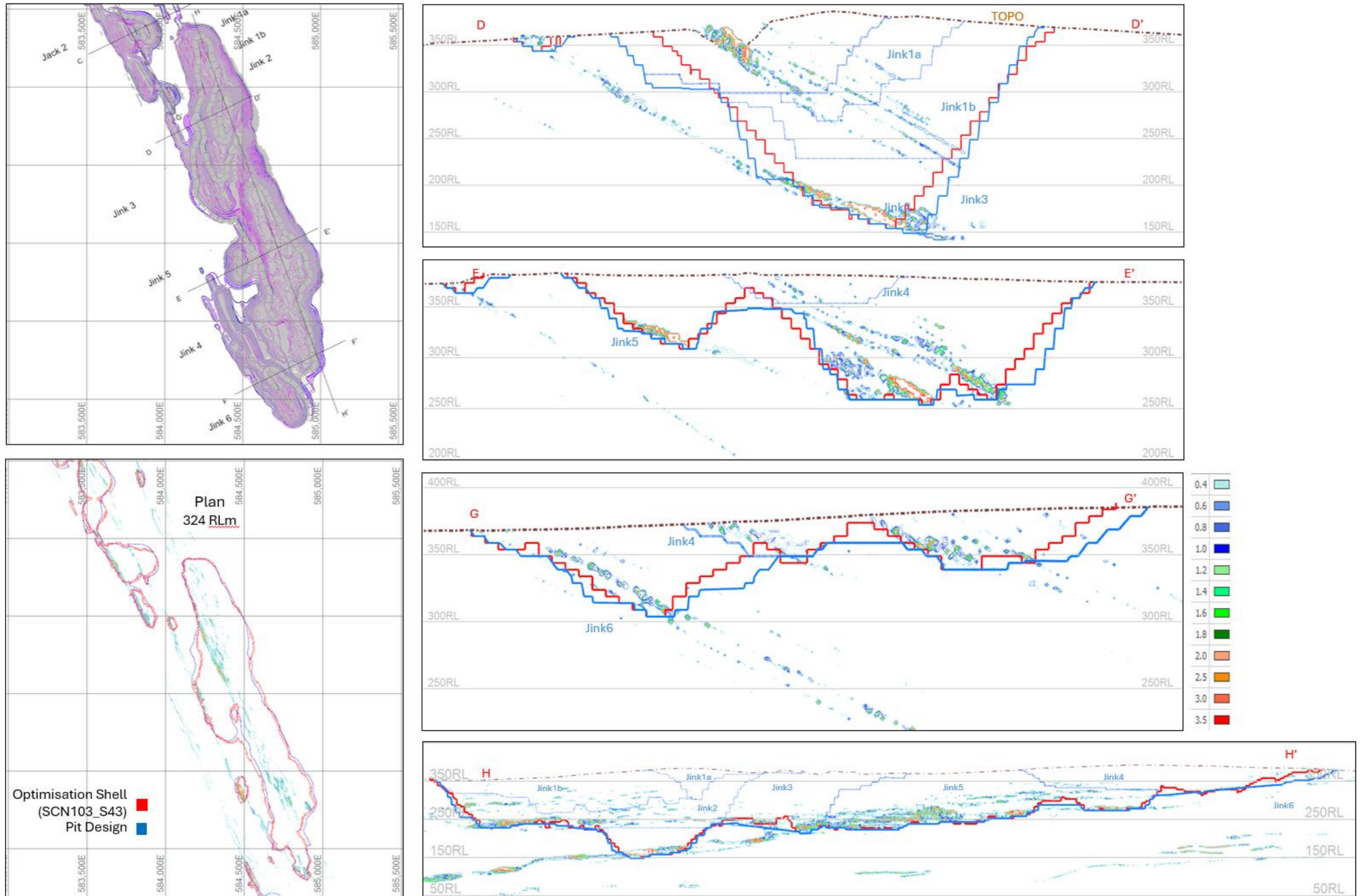


Figure 6-6 Jinkas Designs versus Optimisation Shell – Plan and Sections



### 6.2.3 Jackson and Olympia Mine Area

The northern section of the Central Zone consists of four pits in Jackson and five in Olympia (Figure 6-7). The largest, Jack 1, reaches a depth of approximately 110 m. In contrast, the Olympia pits are relatively small, with average depths ranging from 25 to 40 m. Jack 3, Jack 4, and part of Olympia 4 are intersected by Warren Road, requiring a specific road diversion strategy, which is discussed further in section 6.5.2. These pits, situated along Warren Road, are approximately 5 km from the process plant ROM pad.

The Jackson and Olympia designs captured an additional 7,500 oz from 5.8% more ore at an average grade of 0.80 g/t at the cost of 14% additional waste resulting in an increase in strip ratio from 6.2:1 to 6.7:1. The shell comparison results are presented in Table 6-5.

**Table 6-5 Jackson & Olympia Shell to Design Comparison**

Design vs Shell	Material	Grand Total			
		Volume (Mbcm)	Mass (Mt)	DAU (g/t)	Au (koz)
Jackson and Olympia	Total Ore	2.1	5.3	1.1	185.8
	WASTE	14.5	36.1		
	Total Rock	16.6	41.5		
Optimisation Shell SCN103_S43	Total Ore	2.0	5.1	1.1	178.3
	WASTE	12.7	31.6		
	Total Rock	14.7	36.6		
<b>Difference (%)</b>	<b>Total Ore</b>	<b>5.3%</b>	<b>5.8%</b>	<b>-1.5%</b>	<b>4.2%</b>
	<b>Waste</b>	<b>14.1%</b>	<b>14.4%</b>	<b>0.0%</b>	

Figure 6-8 presents a comparison of the pit design against the optimal shell, including the plan view and selected cross-sections.

Figure 6-7 Pit Design – Jackson and Olympia Mine Area

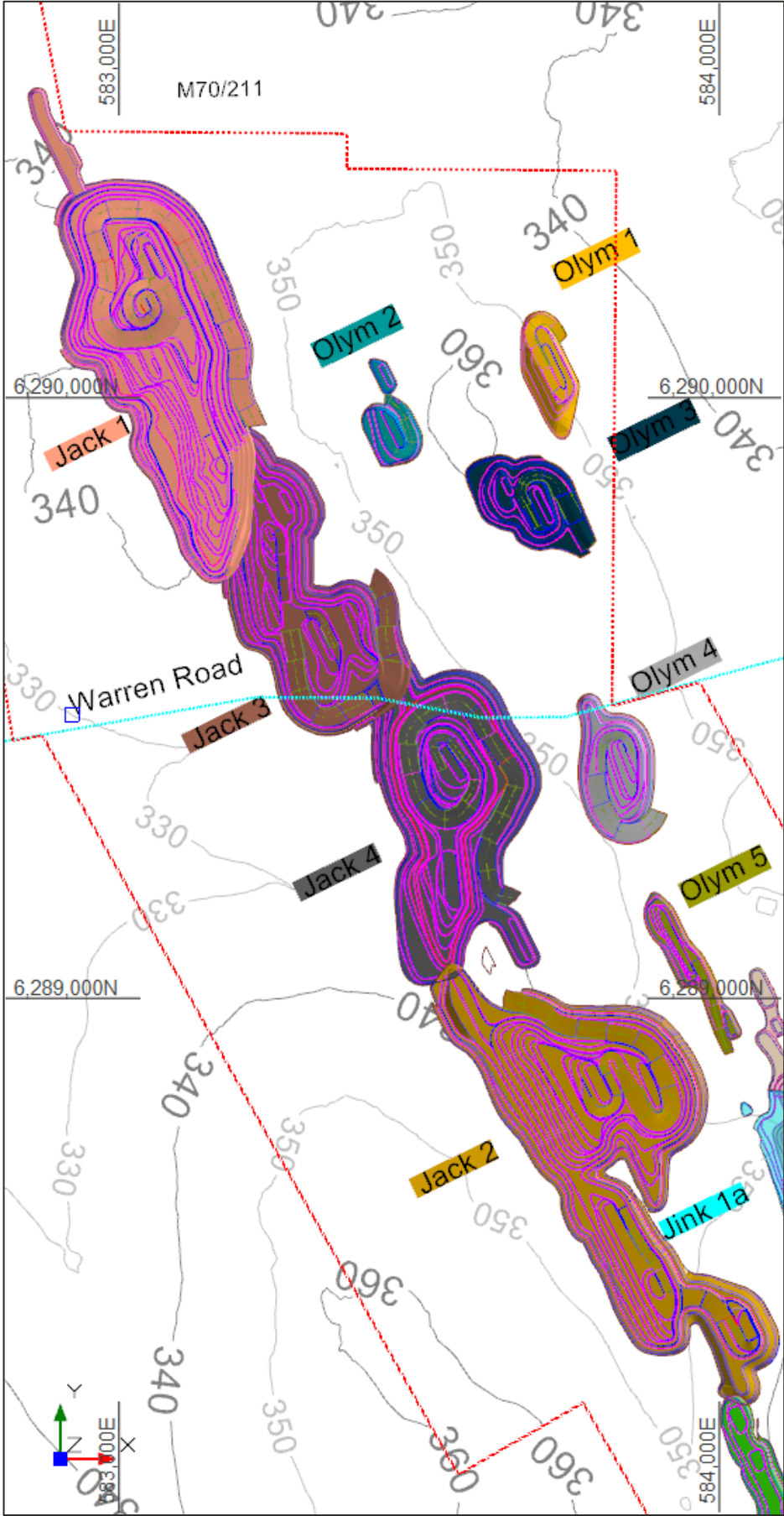
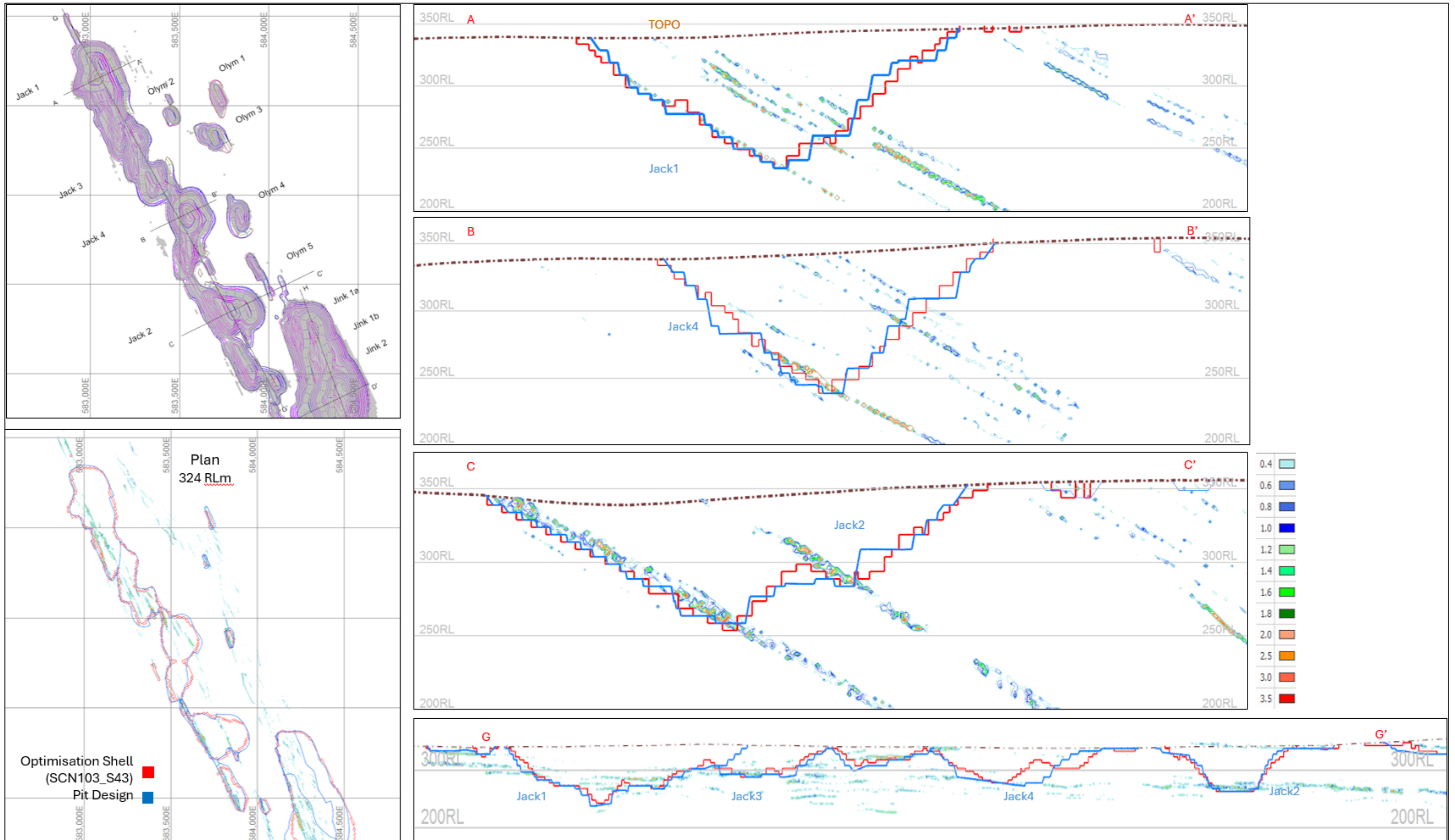


Figure 6-8: Jackson and Olympia Designs versus Optimisation Shell – Plan and Sections



## 6.2.4 Dingo Pit Design

The Dingo pits are located adjacent to the Rifle Range exclusion zone to the south, approximately 4 km from the process plant ROM pad. Due to this proximity, the ore does not require stockpiling or rehandling by road train. The South Zone deposit consists of two pits, as shown in Figure 6-9. The larger of the two, Dingo 1, is situated to the north and reaches a depth of approximately 140 m, while the smaller pit, Dingo 2, is 80 m deep.

The Dingo designs captured and additional 2,200 oz from 3.3% more ore at an average grade of 0.74 g/t at the cost of 4% additional waste resulting in an increase in strip ratio from 7.8:1 to 7.9:1. The shell comparison results are presented in Table 6-6.

**Table 6-6 Dingo Shell to Design Comparison**

Design vs Shell	Material	Grand Total			
		Volume (Mbcm)	Mass (Mt)	DAU (g/t)	Au (koz)
Dingo Design	Total Ore	1.2	2.9	1.0	91.5
	WASTE	9.9	22.8		
	Total Rock	11.1	25.6		
Dingo SCN206_S42	Total Ore	1.1	2.8	1.0	89.3
	WASTE	9.4	21.8		
	Total Rock	10.6	24.5		
Difference	<b>Total Ore</b>	<b>3.2%</b>	<b>3.3%</b>	<b>-0.8%</b>	<b>2.4%</b>
	<b>Waste</b>	<b>5.4%</b>	<b>4.3%</b>	<b>0.0%</b>	

Figure 6-10 presents a comparison of the design against the optimal shell, including the plan view and selected cross-sections.

Figure 6-9 Pit Design – Dingo Mine Area

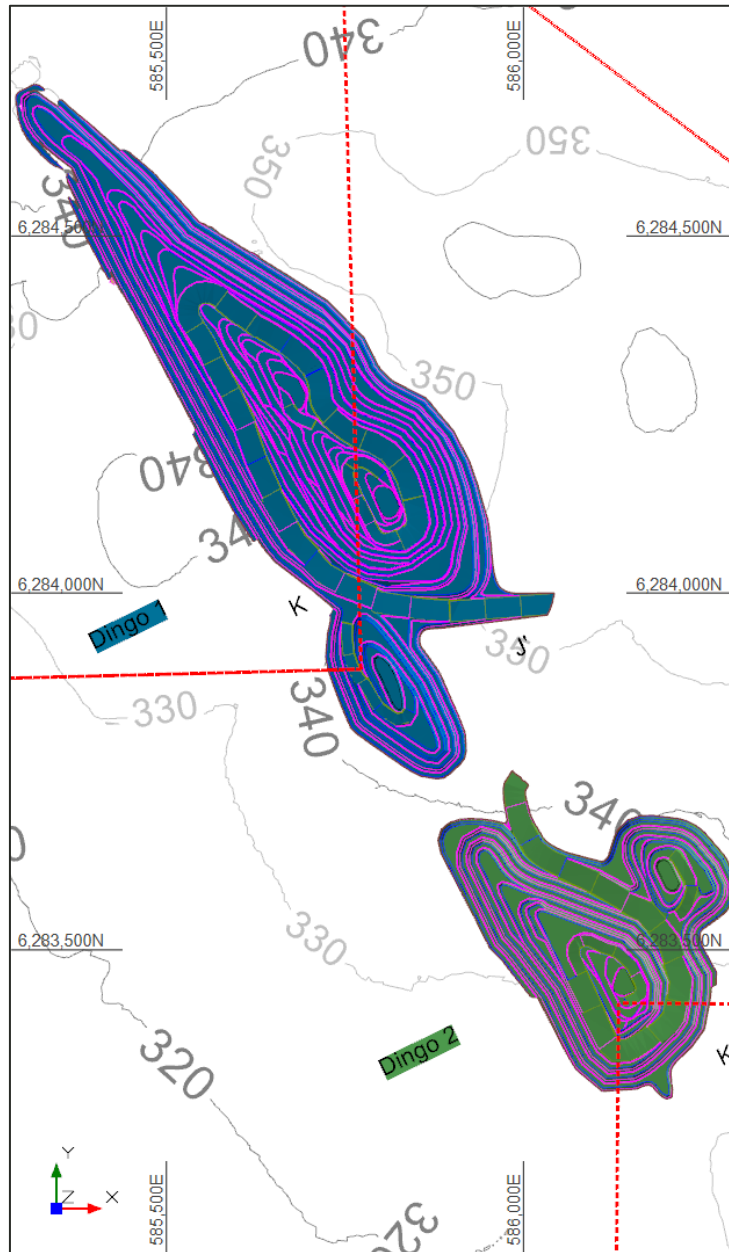
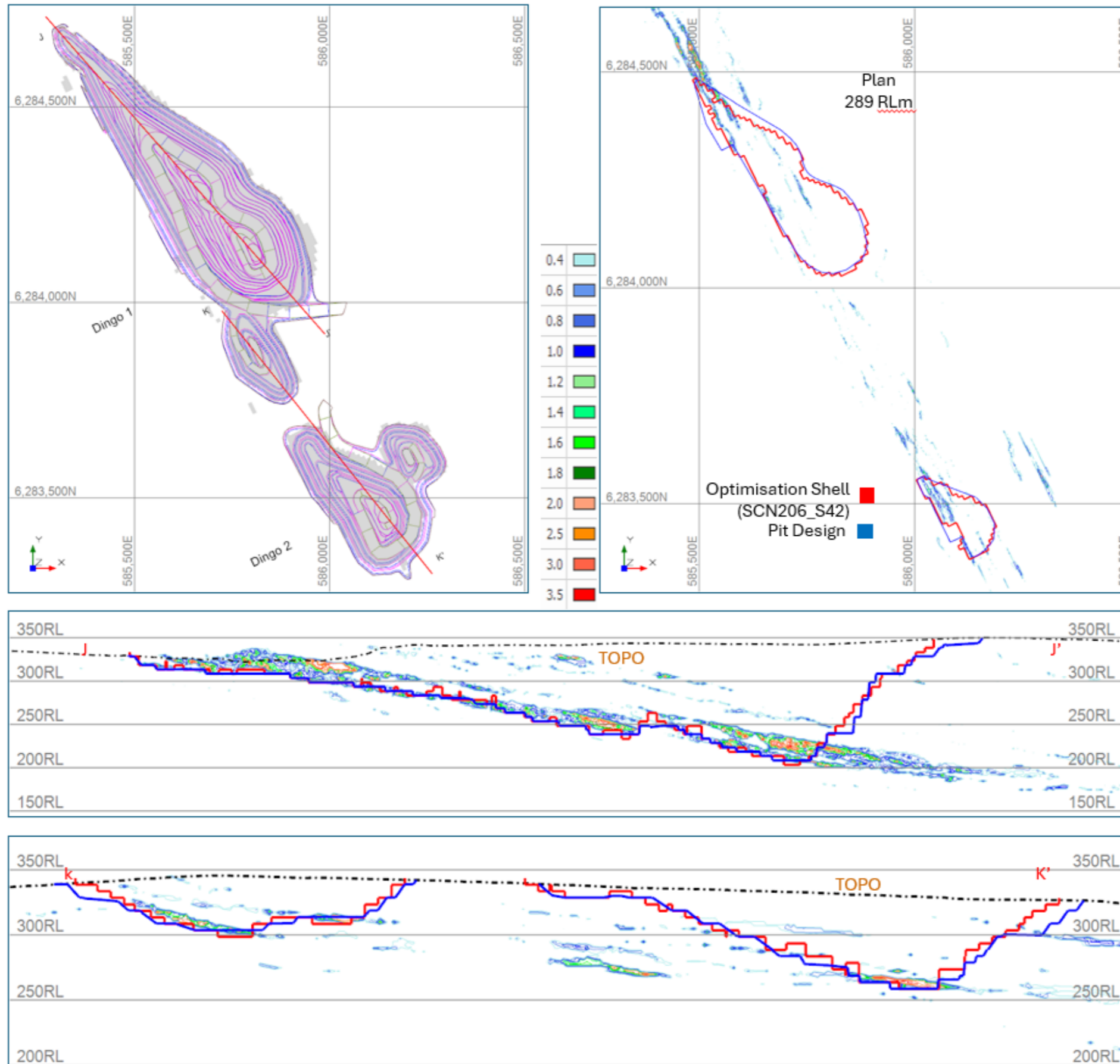


Figure 6-10 Dingo Designs versus Optimisation Shell – Plan and Sections



### 6.3 Stage Inventories

The final designs contain a total of 1,258 koz (+6.6%) from 35.3 Mt of ore (+7.8%) at 1.11 g/t (-1.1%) at a strip ratio of 6.9:1 (+3.6%). This includes Inferred material totalling 159 kt at 0.99 g/t. Table 6-7 summarises the mine inventory by stage.

Table 6-8 summarises the quantity of Inferred by pit phase which makes up less than 0.5% of the total inventory. The inferred material is included in the mine schedule.

**Table 6-7: Inventories by Option and by Stage**

Phase	Total Ore			Waste	Total Rock	SR
	Tonnes	DAU (g/t)	Ounces	Tonnes	Tonnes	
jack1	1,821,894	1.19	69,420	12,468,925	14,290,819	6.8
jack2	1,562,241	0.90	45,296	8,772,475	10,334,716	5.6
jack3	741,993	1.11	26,507	4,568,958	5,310,951	6.2
jack4	711,043	1.29	29,411	8,461,608	9,172,651	11.9
Jink1	5,592,610	1.33	238,631	30,045,758	35,638,369	5.4
Jink2	2,332,132	1.22	91,504	26,154,902	28,487,034	11.2
Jink3	5,075,044	1.44	234,891	41,719,369	46,794,414	8.2
Jink4	3,619,438	0.83	96,331	26,891,917	30,511,354	7.4
Jink5	9,002,273	0.97	282,156	50,466,730	59,469,003	5.6
Jink6	1,500,295	0.78	37,627	8,040,163	9,540,458	5.4
olym1	110,797	0.70	2,505	328,599	439,396	3.0
olym2	60,315	1.20	2,325	110,605	170,920	1.8
olym3	102,478	0.90	2,955	510,276	612,754	5.0
olym4	195,594	1.00	6,295	628,746	824,340	3.2
olym5	38,370	0.85	1,043	258,632	297,001	6.7
ding1	2,304,158	1.02	75,708	18,525,792	20,829,950	8.0
ding2	566,460	0.86	15,624	4,186,881	4,753,341	7.4
<b>Total</b>	<b>35,337,135</b>	<b>1.11</b>	<b>1,258,229</b>	<b>242,140,336</b>	<b>277,477,471</b>	<b>6.9</b>

**Table 6-8 Inferred within Pits**

Quantity	ding2	jack4	Jink2	Jink3	Jink6	olym3	olym4	olym5	Grand Total
Tonnes	15,747	72,054	9,534	54,263	5,117	132	719	1,276	158,842
DAU (g/t)	0.76	1.23	0.64	0.81	0.77	0.95	0.85	0.91	0.99
Ounces	386	2,855	196	1,419	127	4	20	37	5,044
Proportion of Ore (t)	2.8%	10.1%	0.4%	1.1%	0.3%	0.1%	0.4%	3.3%	1.5%

## 6.4 Waste Dump Design

The Waste Rock Dump (WRD) designs have been developed with based on the as-mined waste volumes and the following assumed swell factors:

- Oxide / Transitional = 20%
- Fresh = 25%

WRD slopes design criteria assumed a stepped final rehabilitation surface with a maximum dump height of 80 m above ground level. The waste dump has been built with 18° batter and 8 m wide berms to give an overall slope of 16° with 20.0 m high lifts as outlined in Figure 6-11 for the Central zone and Figure 6-12 for the South zone.

Figure 6-11 Ultimate Pit and Waste Dump – Central Layout

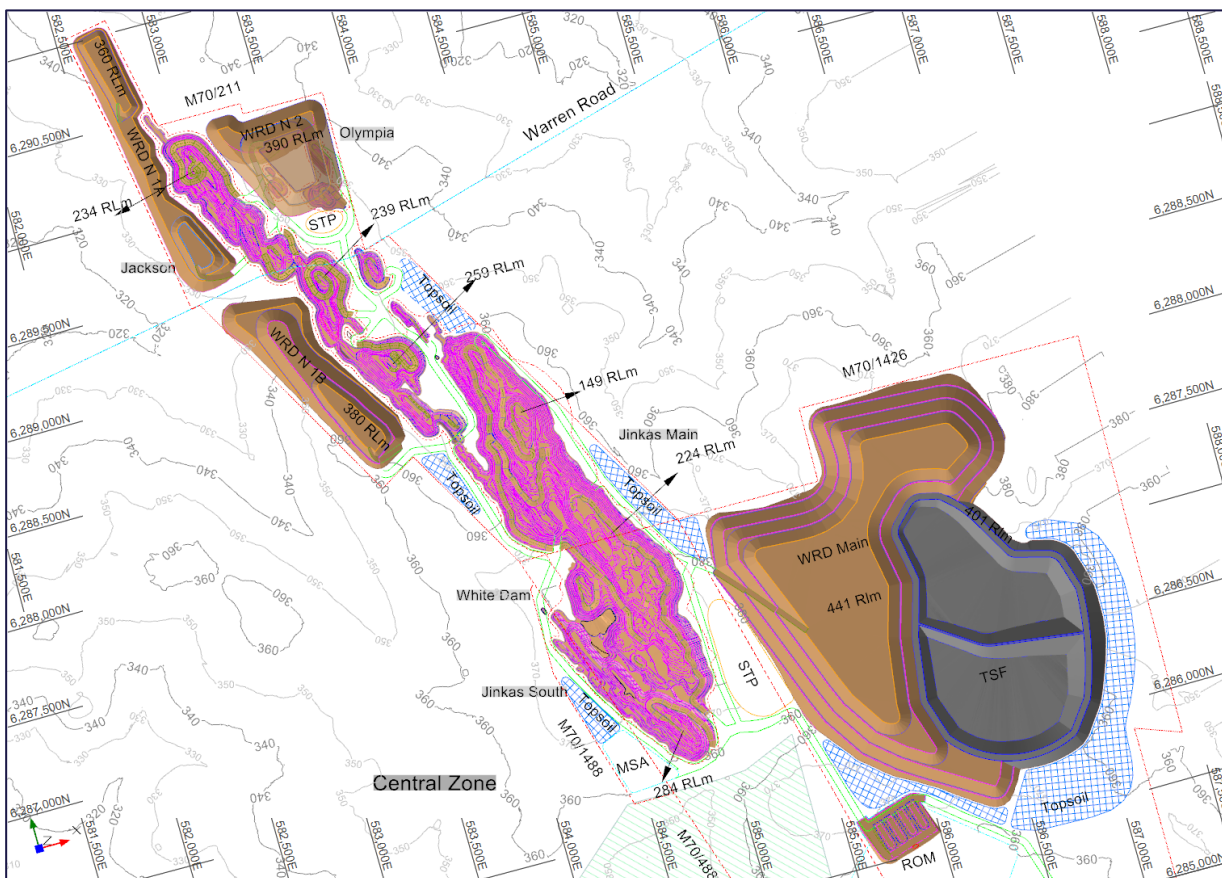


Figure 6-12 Ultimate Pit and Waste Dump – South Zone Layout



All waste from the Jinkas Pits is directed to the construction of the tailings dam and the main waste dump. Approximately 30% of the waste from Jackson is transported to the main waste dump, while the remaining portion is directed to the northern waste dumps.

For the Olympia Pits, all waste is allocated to Waste Dump North 2, Stage 1. Upon the completion of mining activities in the Olympia Area, Stage 2 of this dump will be constructed. The development of Stage 2 requires backfilling the Olympia Pits before additional waste is placed on top. Additionally, Waste Dumps North 1 and 2 are designed as separate dumps to ensure that they do not obstruct Warren Road.

In the South zone, two waste dumps and a backfill have been designed to manage all the waste. The backfilling of the northeast section of Dingo 1 pit is intended to minimize the zone of instability, shifting it from the woodland areas to the pit crest. Table 6-9 outlines the volume required as well as where this material will be placed.

Table 6-9: Waste dump and Backfill Statistics

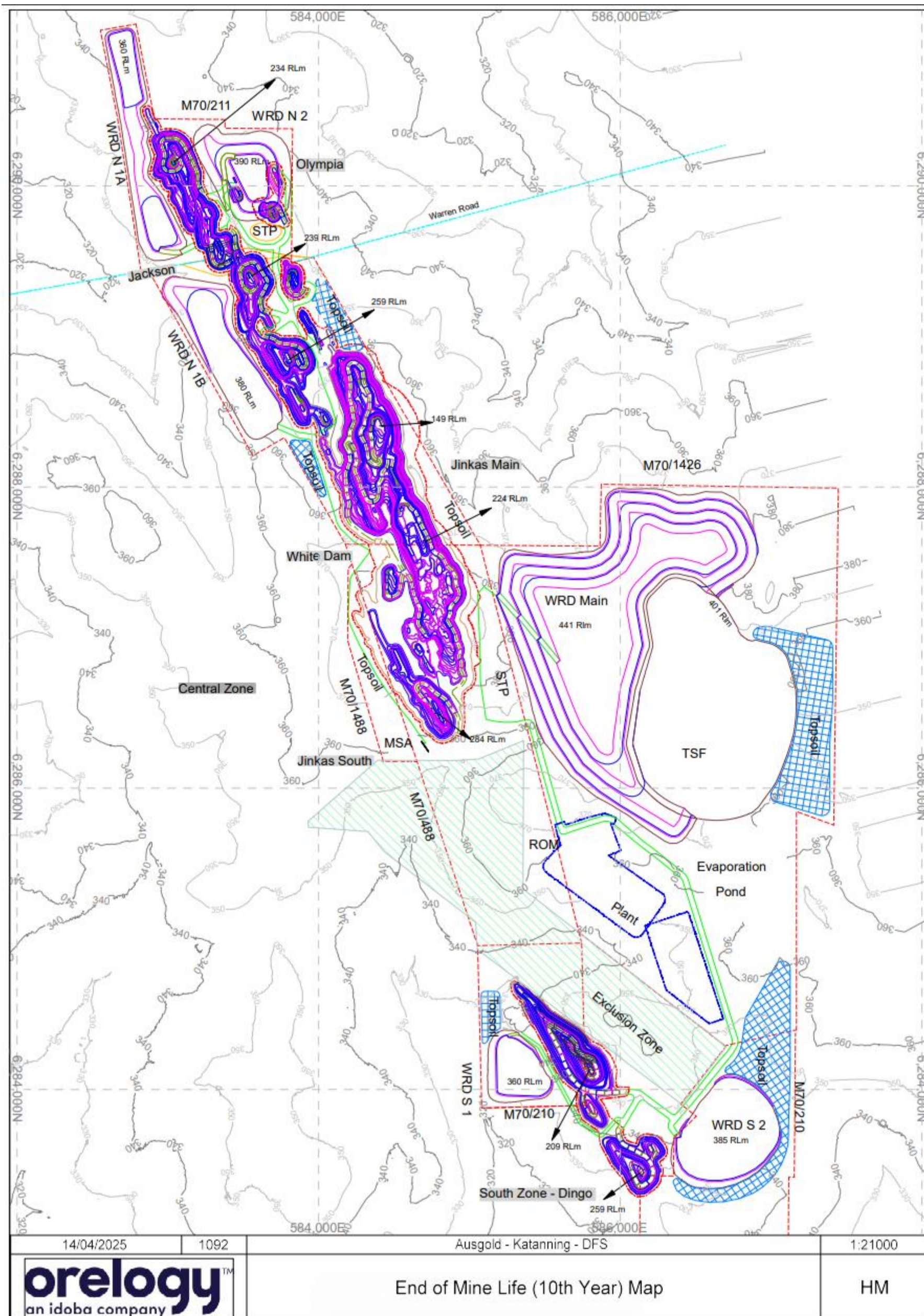
Source/Destination	Area	Weathering	Total Waste (Mlcm)
Volume Excavated (Source)	Jackson	OX/TR	11.4
		FR	5.1
		<b>Total</b>	<b>16.6</b>
	Jinkas	OX/TR	38.1
		FR	50.1
		<b>Total</b>	<b>88.2</b>
	Olympia	OX/TR	1.1
		FR	0.0
		<b>Total</b>	<b>1.1</b>
	Dingo	OX/TR	8.6
		FR	3.5
		<b>Total</b>	<b>12.1</b>
<b>Total</b>			<b>117.9</b>
WRD Design (Destination)	Jackson (North1A)	<b>Total</b>	3.4
	Jackson (North1B)	<b>Total</b>	6.1
	Jackson (North2 STG1)	<b>Total</b>	2.2
	Jackson (North2 STG2 + Backfill)	<b>Total</b>	5.0
	Jinkas ( Main WRD)	<b>Total</b>	79.3
	Jinkas (TSF)	<b>Total</b>	14.7
	Jinkas3 (Backfill)	<b>Total</b>	0.6
	Dingo (Main WRD )	<b>Total</b>	11.1
	Dingo (North WRD )	<b>Total</b>	2.3
	Dingo (Backfill)	<b>Total</b>	1.4
<b>Total</b>			<b>126.0</b>
Difference (%)	Jackson and Olympia	<b>Total</b>	-6%
	Jinkas	<b>Total</b>	6%
	Dingo	<b>Total</b>	23%
	<b>Total</b>		

## 6.5 General Site Layout

The majority of the mine infrastructure (i.e. Process facility) will be concentrated south of the Jinkas Pits and east of the Rifle Range exclusion zone. All Jinkas and Dingo Pits will be connected to the processing facility and stockpiling areas via haul roads.

Offices and workshops will be located southwest of the Jinkas Pit (Stage 6). Figure 6-13 provides the proposed layout and location.

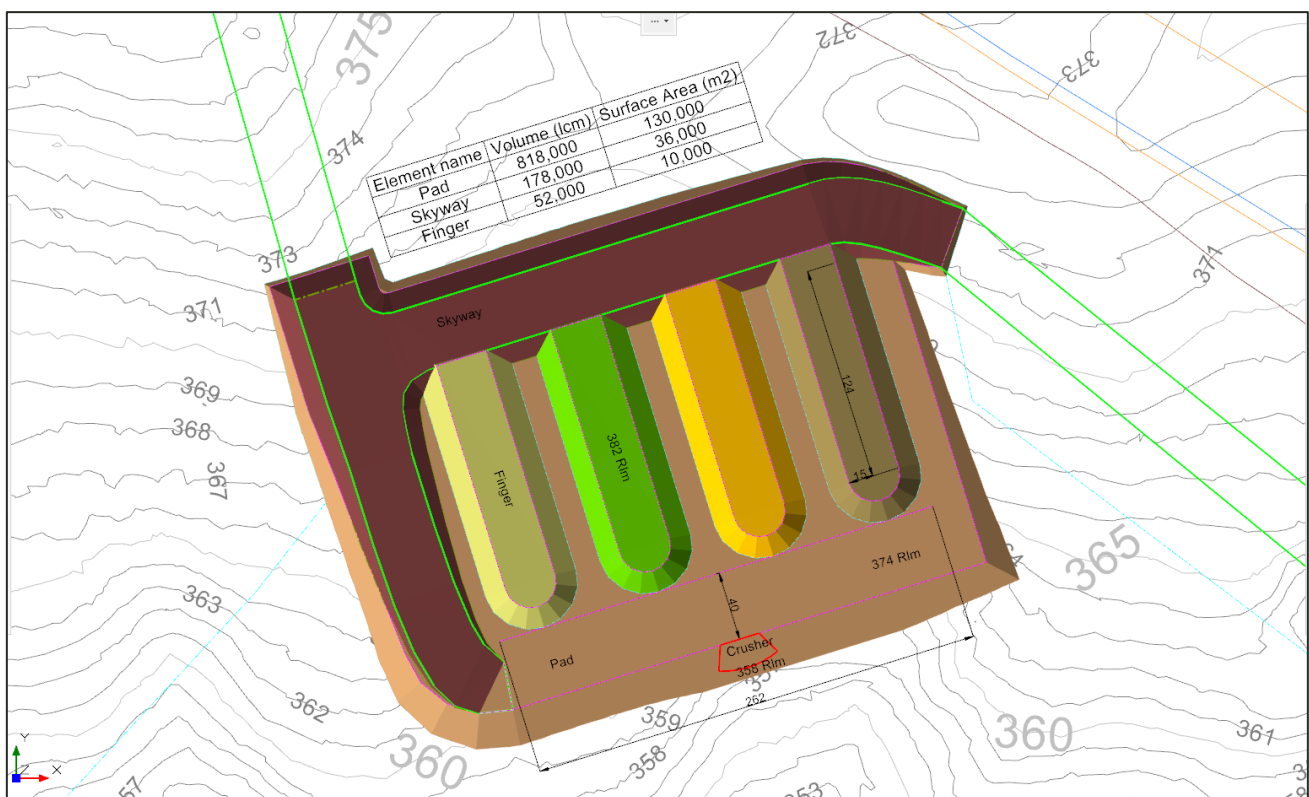
Figure 6-13: Proposed Layout and Location



## 6.5.1 ROM Design

The main ROM pad and processing facility will be located south of the Southern Zone, between the waste dump and the Rifle Range. The ROM pad is designed to accommodate both direct tipping via haul trucks into the crusher for ore reclaimed from stockpile and sufficient storage capacity from the skyway for feed using a Front-end loader (see Figure 6-14). It is elevated 16 metres above the natural topography to align with the assumed height of the crusher feed bin. An 8-metre skyway sits atop the ROM pad, allowing haul trucks to dump material onto fingers.

Figure 6-14 ROM Pad Design



To manage blending, the ROM has been designed with four fingers each of 52,000 lcm capacity, which equates to approximately 80 kt for an oxide finger or 120 kt for a mixed transitional and fresh finger. At less than 5% of the feed, it will be possible to store a small quantity of the “reactive” ore on the ROM without a dedicated finger.

The multiple finger designs provide opportunity to construct, close and then completely feed a finger for the geology team to reconcile the ore feed to the plant. With all fingers filled, the ROM pad will have a capacity of 440 kt which equates to more than 6 weeks of ore feed at 3.6 Mtpa.

## 6.5.2 Warren Road Diversion Strategy

Ausgold intend to maintain traffic flow on the Warren Road as it is a key thoroughfare for farmers in the region. The strategy to keep the road open is as follows:

**Phase 1** – Mine out the Olympia pits (1,2 & 3) that lie north of the Warren road. This is necessary to provide adequate area for waste dump storage from Jackson Stages 1 and 3. A controlled intersection at Warren road will be required on dayshift only for ore haulage. Any ore mined on night shift will be placed on stockpile for rehandle during daylight hours. All waste will be placed on the WRD\_N2 waste dump directly north of the pits. This will occur in Year 3.

**Phase 2** – Mine out Jackson Stage 1 (Jack 1) with waste placed in the WRD\_N1 waste dump to the west or the WRD\_N2 waste dump to the east of Jack 1 pit. As per phase 1, a controlled intersection at Warren road will be required on dayshift only for ore haulage and any ore mined on night shift will be placed on stockpile for rehandle during daylight hours (Figure 6-15). This will occur in Years 3 and 4.

**Phase 3** – Warren road is diverted to the south of Jackson stage 2 (Jack 2) and north of Olympia Stage 4 (Figure 6-16). This road can be constructed while the existing road is still active to minimise disruption to public road access. Once the road is diverted, mine out Jackson Stage 3 immediately to the south of Jack 1. Waste from this pit will be sent to the extended WRD\_N2 which backfills Olympia pits 1, 2 & 3. As with Phase 1 & 2, this will require a controlled intersection at Warren road on dayshift only for ore haulage and any ore mined on night shift will be placed on stockpile for rehandle during daylight hours. This will occur in Year 3 after mining of Olympia pits 1 to 3 are completed.

**Phase 4** – Once Jack 3 is completed, the south end will be backfilled with 600 kbcm of waste from Jackson Stage 4. This will be used as a foundation for a new diversion road around top end of Jack 4. This may require the road to be closed for 4 to 6 weeks while backfill is completed. Once the new diversion road is completed, the old diversion road will be closed and mining activity north of the Warren Road ceased (Figure 6-17). This will occur in Year 7 prior to mining Jack 2.

The intersection between Warren Road and the haul road leading to the ROM will require traffic lights or a similar crossing solution to ensure safe and efficient traffic flow.

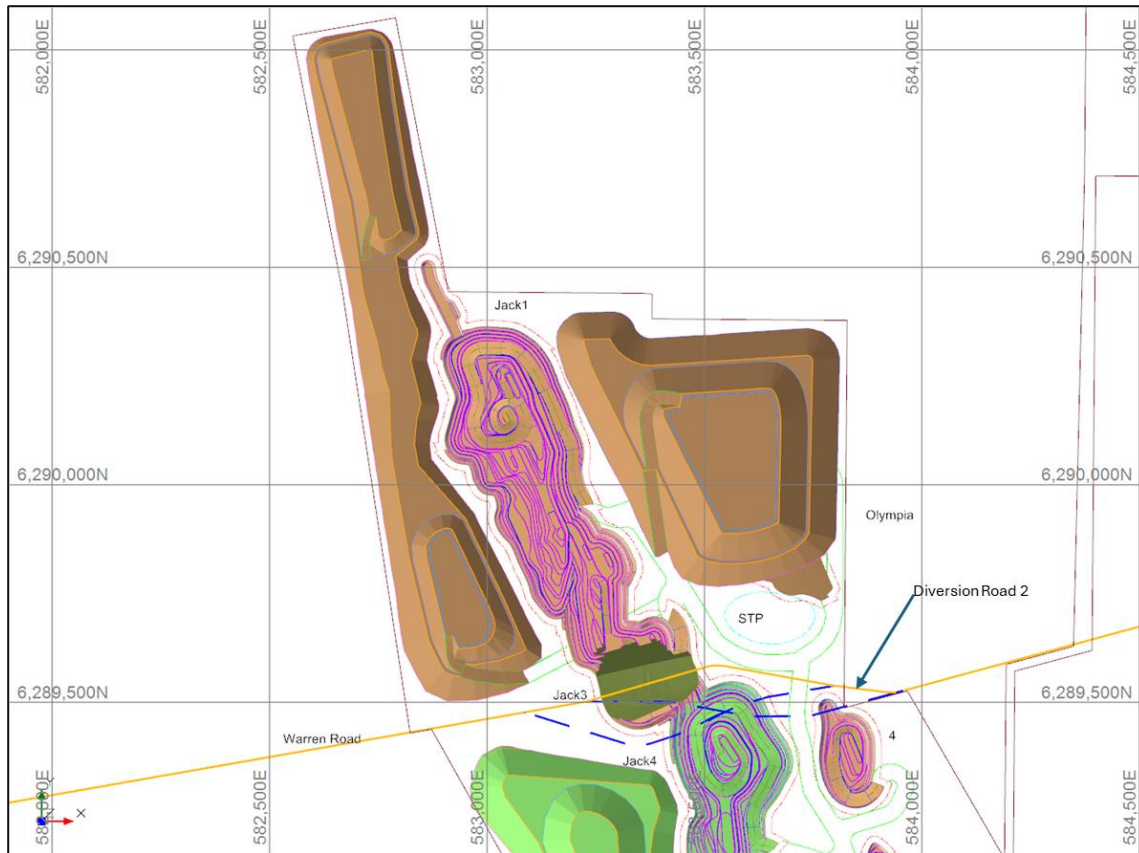
Figure 6-15 Location of Traffic control during Phase 1 and Phase 2



Figure 6-16 Phase 3 Warren Road Diversion 1



Figure 6-17 Phase 4 Final Warren Road Diversion



## 7 Mine Production Schedule

The purpose of the scheduling activity was to generate a practical, realistically achievable schedule that maximises project value within the given constraints and objectives, as discussed below. A practical, realistically achievable schedule:

- Satisfies mill feed requirements.
- Incorporates ramp-up of operations for mining and processing.
- Minimises any pre-strip and generally delaying waste mining where possible.
- Avoids excessive vertical advance rates.
- Mines from a realistic number of areas per period.
- Schedules achievable material movements per area.
- Minimises rehandling and keeps stockpile sizes as low as possible.

The inventories used for LOM scheduling are outlined in Table 6-7.

### 7.1 Scheduling Method

The Katanning Project pits were scheduled using Hexagon MinePlan Schedule Optimizer™ (MPSO), an advanced mine scheduling tool that optimizes cut-off grades, phase sequencing, and haulage while considering operational constraints. MPSO integrates multiple objectives to deliver practical and efficient schedules that maximize project value.

MPSO offers key functionalities such as:

- **Optimized Phase Scheduling:** Determines the best extraction sequence to maximize Net Present Value (NPV) while meeting operational constraints.
- **Cut-off Grade Optimization:** Adjusts cut-off grades dynamically to enhance profitability over the life of the mine.

This approach ensures that the scheduling process aligns with project objectives, delivering an optimal production plan. The mining model was limited to all blocks that were fully or partially within the ultimate pit designs and assigned with corresponding proportions and a “Stage Identifier”. The ore inventory was then reported from the MineSight™ general mining package directly to MPSO.

## 7.2 Scheduling Objectives and Constraints

### 7.2.1 Scheduling Periods

The schedule assumes that the mining contractor will mobilize during the two quarters before plant production begins, with initial stripping, clearing, and pre-strip-mining taking place in the three months leading up to plant production. In the initial version of the schedule, the pre-strip period was set at four months to provide transitional and fresh material for the construction of the tailings dam. In order to reduce initial project capital, the final schedule was adjusted back to three months resulting in the first month of operations starting in the final month of Y0. Material brought forward to meet the additional plant feed tonnes were taken from stockpile.

Consequently, the schedule was run as outlined below and then reported with three months of pre-production after adjusting the stockpile reclaim timing:

- 4 x monthly periods (Pre-production).
- 24 x monthly periods (i.e. 2 years).
- 16 x quarterly periods (i.e. 4 years).
- annual periods thereafter.

The periods are detailed in Table 7-1.

Table 7-1 Scheduling Periods

Period #	Year ID	Period label	Duration	Period #	Year ID	Period label	Duration	Period #	Year ID	Period label	Duration	
			Months				Months				Months	
1	Y0	1>Y0M-4	1	17	Y2	17>Y2M13	1	33	Y4	33>Y4Q13	3	
2		2>Y0M-3	1	18		18>Y2M14	1	34		34>Y4Q14	3	
3		3>Y0M-2	1	19		19>Y2M15	1	35		35>Y4Q15	3	
4		4>Y0M-1	1	20		20>Y2M16	1	36		36>Y4Q16	3	
5	Y1	5>Y1M1	1	21	Y2	21>Y2M17	1	37	Y5	37>Y5Q17	3	
6		6>Y1M2	1	22		22>Y2M18	1	38		38>Y5Q18	3	
7		7>Y1M3	1	23		23>Y2M19	1	39		39>Y5Q19	3	
8		8>Y1M4	1	24		24>Y2M20	1	40		40>Y5Q20	3	
9		9>Y1M5	1	25		25>Y2M21	1	41	Y6	41>Y6Q21	3	
10		10>Y1M6	1	26		26>Y2M22	1	42		42>Y6Q22	3	
11		11>Y1M7	1	27		27>Y2M23	1	43		43>Y6Q23	3	
12		12>Y1M8	1	28		28>Y2M24	1	44		44>Y6Q24	3	
13		13>Y1M9	1	29		Y3	29>Y3Q9	3	45	Y7	45>Y7	12
14		14>Y1M10	1	30			30>Y3Q10	3	46	Y8	46>Y8	12
15		15>Y1M11	1	31			31>Y3Q11	3	47	Y9	47>Y9	12
16		16>Y1M12	1	32			32>Y3Q12	3	48	Y10	48>Y10	12

## 7.2.2 Stage Release

To facilitate scheduling, the ultimate pit was split into eighteen stages to ensure ore continuity at reasonable strip ratios. Figure 7-1 shows the staging of the pits for the Central and South zones utilised for the LOM Production Schedule. It should be noted Jinkas Stage 1 is split into a west (Stage1A) and east (Stage1B) to aid in balancing mining and maintaining ore supply over the initial monthly periods.

Figure 7-1 Pit Stage Layout for LOM Production Schedule



## 7.2.3 Workdays per Year

Orelogy included an allowance for eighteen days per annum of production interruptions due to the weather. In addition, two more days have been added as result of other interruptions (public holidays, etc). Hence mining activities are restricted to 345 days per year.

## 7.2.4 Shifts and Shift Duration

The mining roster was based on two 12 hour shifts per day operating seven (7) days per week.

# 7.3 Production Schedule Assumptions

## 7.3.1 Mining Targets and Constraints

Mine scheduling requires balancing conflicting objectives and constraints. For the Katanning Project, the primary goal is to maximize Net Present Value (NPV) while adhering to practical mining limitations. The key considerations include:

- Conducting pre-production mining to produce approximately 5 Mt of total transitional and fresh material for tailings dam construction. The oxide waste will be used to construct the ROM pad.
- Prioritizing high-grade material extraction in the early years.
- Maintaining a production rate of more than 130 koz/year for the first four years to maximise and smoothing cash flow during the payback period.
- Deferring mining in the Jackson area until the end of Year Three to delay the capital cost for diversion of the Warren Road.
- Limiting oxide content to a maximum of 25% per period, as laboratory results indicate processing issues when exceeding this threshold.
- Restricting Jackson Stage 1 ore to no more than 8% per period, due to its higher processing cost, lower recovery rate, and increased risk associated with reactive pyrrhotite.
- Enforcing a maximum bench turnover rate per pit equivalent to 80 m vertical advance per year, as follows:
  - 2 per month.
  - 4 per quarter.
  - 16 per year.

This approach ensures an optimized balance between economic returns, operational efficiency, and practical constraints

### 7.3.2 Processing

The metallurgists provided variable feed rates for the processing plant dependent on the mining area and the weathering conditions of the rock. The schedule assumes 8,000 operating hours per year for the crusher and converts the oxide, transitional and fresh throughput rates into an hourly rate, as outlined in Table 7-2. The throughput rates used in the schedule are 4.1 Mtpa for oxide material and 3.8 Mtpa for trans material. For fresh material, the rates are 3.6 Mtpa in the North Zone and 3.4 Mtpa in the South Zone.

Table 7-2 Throughput rate (tph)

Mining Area	North			South zone		
	Jinkas, Jackson, and Olympia			Dingo		
Weathering	Oxide	Trans	Fresh	Oxide	Trans	Fresh
Tonnes per hour	517.7	476.0	450.0	517.7	476.0	424.2
Mt/year equivalent	4.1	3.8	3.6	4.1	3.8	3.4

A plant ‘ramp up’ schedule was also provided which equated to the following ramp up over the first four months of the schedule as shown in Table 7-3.

Table 7-3 Plant Ramp up

Y1M1	Y1M2	Y1M3	Y1M4 and onwards
50%	83%	96%	100%

### 7.3.3 Mining Productivity

Equipment productivity for the mining fleet was based on 85% availability and 85% utilisation for the operating 345 days resulting in 6,193 total available hours. The production rates for each loading unit by material type is shown in Table 7-4. The mining schedule assumes that each loading unit always has enough trucks to operate at the modelled capacity.

Table 7-4 Loading Productivity Assumptions

Description	Units	Ox_WST	Tr_WST	Fr_WST	Ox_ORE	Tr_ORE	Fr_ORE
Density	t/m <sup>3</sup>	1.80	2.71	2.81	1.80	2.74	3.10
Loading Unit		EX2600			EX1200		
Truck (excl. Fuel)		785	785	785	777	777	777
Loading Unit bucket size	m <sup>3</sup>	17.0	17.0	17.0	7.0	7.0	7.0
Bucket fill factor	%	99%	90%	85%	95%	90%	85%
Calculated Max. Bucket Capacity	m <sup>3</sup>	16.8	15.3	14.5	6.7	6.3	6.0
Swell factor	%	15%	20%	20%	15%	15%	20%
Moisture content	%	5%	1%	1%	5%	1%	1%
Loose Wet Density	wmt/m <sup>3</sup>	1.64	2.28	2.37	1.64	2.41	2.61
Rated Lift	t	30.6	30.6	30.6	12.6	12.6	12.6
Calculated Lift	t	27.7	34.9	34.2	10.9	15.2	15.5
Possible Bucket Payload	wmt	27.7	30.6	30.6	10.9	12.6	12.6
	m <sup>3</sup>	16.8	13.4	12.9	6.7	5.2	4.8
Digger De-rating Factor	%	90%	100%	100%	87%	100%	100%
Bucket Cycle Time (higher for hard rock)	seconds	30.00	34.00	34.00	30.00	34.00	34.00
Tray Fill Factor	%	100%	100%	100%	100%	100%	100%
Dump Truck Rated Capacity (incl. FF)	m <sup>3</sup>	85.0	85.0	85.0	60.0	60.0	60.0
Dump Truck Rated Capacity	t	141.0	141.0	141.0	91.7	91.7	91.7
Max. Dump Truck Capacity	wmt	139.7	141.0	141.0	91.7	91.7	91.7
Passes per truck theor.	#	5.1	4.6	4.6	8.4	7.3	7.3
Actual # Passes	#	6.0	5.0	5.0	9.0	8.0	8.0
Final Truck Payload	wmt	139.7	141.0	141.0	91.7	91.7	91.7
	dmt	133.0	139.6	139.6	87.3	90.8	90.8
First Bucket Drop Time	minutes	10.00	10.00	10.00	10.00	10.00	10.00
Loading spot time	minutes	20.00	20.00	20.00	20.00	20.00	20.00
Total load Time	seconds	180	170	170	270	270	270
Job efficiency factor (Effective minutes/hour)	%	85%	85%	85%	85%	85%	85%
Loading Unit Productivity (Incl. Efficiency Factor)	wmt / Op. Hr	2,375	2,538	2,538	1,039	1,039	1,039
	dmt / Op. Hr	2,262	2,513	2,513	990	1,029	1,029
	SMU Factor	90.0%	90.0%	90.0%	88.0%	88.0%	88.0%
	dmt / Eng. Hr	2,036	2,262	2,262	871	905	905
Derating for under-trucking (Based on round down < 25% & round up for > 25% of truck)	%	96%	94%	94%	91%	90%	90%
Average Productivity (dmt / Eng.hr)	dmt / Eng. Hr	1,944	2,126	2,126	793	815	815
Mining Production per Year	Mt	12.0	13.2	13.2	4.9	5.0	5.0

### 7.3.4 Fleet Levels

Fleet levels were adjusted throughout the schedule as summarised in Table 7-5.

Table 7-5 Mining Rate by Period

From	To	EX1200	EX2600	Mt/period	Mt/year equivalent
0	1>Y0M-4	2	1	1.8	23.1
2>Y0M-3	28>Y2M24	2	2	2.9	35.9
29>Y3Q9	44>Y6Q24	2	2	8.8	35.9
45>Y7	48>Y10	1	1	17.9	17.9

### 7.3.5 Stockpiling Strategy

The stockpiling strategy was primarily based on grade streaming to maximise cashflow with a secondary objective to control the level of oxide material in the ore feed to avoid “flooding” of some processing circuits. A third objective was to limit the amount of reactive transitional material found in Jackson that caused excessive consumption of reagents. To achieve this, the model was developed with a two ore bins and four ore types.

The two ore bins provided a means of separating the high-grade (HG) ore for direct processing from low-grade (LG) ore that could be stockpiled for later use. A cut of 0.8 g/t was used to define HG with LG defined by the breakeven cut-off grade.

The ore types were based on weathering as follows:

- Oxide
- Transitional
- Reactive transitional
- Fresh

### 7.3.6 Warren Road Diversion

The stage ranking shows that Jackson stages 1 and 3 are high value and should be scheduled for mining early in the mine life. However, due to the additional capital and complications of obtaining approvals at the start of the operation, Ausgold elected to delay the commencement of mining these pits until after Year 3.

## 7.4 Production Schedule

A stage ranking system was developed to aid in setting up priorities for scheduling to produce the highest value mine plan. It was based on a Net Smelter return (NRS) value calculation derived from the

Whittle inputs combined with the mining cost for the stage to determine an average value per day as shown in Table 7-6.

Table 7-6 Stage Ranking

Phase	Total Ore				Waste (kt)	Total Rock (kt)	SR	Process Life (YRS)	Stage Value (\$/day)
	Ore (kt)	g/t	koz	NSR/t					
olym2	60	1.20	2.3	76.06	111	171	1.8	0.01	698,020
Jink1	5,593	1.33	238.6	83.31	30,046	35,638	5.4	1.52	556,567
Jink3	5,075	1.44	234.9	91.99	41,719	46,794	8.2	1.40	484,502
olym4	196	1.00	6.3	55.14	629	824	3.2	0.05	371,594
jack1	1,822	1.19	69.4	67.78	12,469	14,291	6.8	0.49	320,563
jack3	742	1.11	26.5	63.28	4,569	5,311	6.2	0.19	314,621
Jink5	9,002	0.97	282.2	51.34	50,467	59,469	5.6	2.50	209,852
olym3	102	0.90	3.0	48.60	510	613	5.0	0.03	200,336
ding1	2,304	1.02	75.7	54.89	18,526	20,830	8.0	0.61	182,208
Jink2	2,332	1.22	91.5	73.98	26,155	28,487	11.2	0.65	153,540
jack4	711	1.29	29.4	75.33	8,462	9,173	11.9	0.19	149,982
jack2	1,562	0.90	45.3	43.03	8,772	10,335	5.6	0.42	131,419
olym1	111	0.70	2.5	28.22	329	439	3.0	0.03	109,354
olym5	38	0.85	1.0	40.73	259	297	6.7	0.01	62,472
ding2	566	0.86	15.6	43.93	4,187	4,753	7.4	0.14	60,207
Jink6	1,500	0.78	37.6	35.91	8,040	9,540	5.4	0.39	54,772
Jink4	3,619	0.83	96.3	40.16	26,892	30,511	7.4	0.93	26,464
<b>Total</b>	<b>35,644</b>	<b>1.11</b>	<b>1,272</b>	<b>63.43</b>	<b>244,896</b>	<b>280,540</b>	<b>6.9</b>	<b>9.6</b>	<b>279,005</b>

A total of six iterations of the schedule were produced before settling on the final schedule that met all of the objectives and constraints. The production physicals for the final schedule are summarized by year in Table 7-7.

Table 7-7: KGP Mining Schedule

	Items	Units	Total		1	2	3	4	5	6	7	8	9	10
TOTAL Mined	Rock mined Ex-pit	Mt	<b>277.5</b>	9.8	34.5	34.7	35.4	35.1	35.1	31.0	21.6	15.3	12.8	12.2
		Mbcm	<b>109.1</b>	4.4	12.7	12.7	13.4	15.4	13.9	11.4	8.1	5.9	6.2	5.1
	Waste Mined	Mt	<b>242.1</b>	8.7	30.8	31.3	31.0	30.5	32.1	26.4	16.9	12.7	11.3	10.5
		Mbcm	<b>96.3</b>	3.9	11.4	11.6	11.8	13.6	12.8	9.9	6.5	4.9	5.5	4.5
	Strip Ratio	W:O	<b>6.9</b>	7.9	8.3	9.5	6.9	6.6	10.7	5.7	3.6	4.8	7.7	6.3
	Ore Mined	Mt	<b>35.3</b>	1.10	3.72	3.30	4.47	4.62	3.01	4.61	4.71	2.64	1.47	1.67
		Mbcm	<b>12.8</b>	0.4	1.2	1.1	1.6	1.8	1.1	1.6	1.6	0.9	0.7	0.7
Ore Mined - Au Grade	g/t	<b>1.11</b>	1.02	1.24	1.34	1.18	1.18	1.25	0.98	0.94	1.01	0.90	0.95	
Direct Feed	Ore	Mt	<b>26.0</b>		1.8	2.7	3.3	3.4	2.7	3.1	3.6	2.3	1.4	1.7
	Au Grade	g/t	<b>1.28</b>		1.8	1.5	1.4	1.4	1.4	1.2	1.1	1.1	0.9	1.0
	Contained Ounces	koz	<b>1,072</b>		103.7	130.9	150.6	155.4	115.3	119.1	124.2	79.9	41.9	51.1
	Recovery	%	<b>90.8%</b>		92.6%	92.0%	90.8%	90.4%	91.4%	91.4%	91.2%	87.3%	90.5%	86.8%
Stockpiled	Ore	Mt	<b>9.35</b>	1.1	1.9	0.6	1.1	1.2	0.4	1.5	1.1	0.3	0.0	
	Au Grade	g/t	<b>0.62</b>	1.02	0.74	0.56	0.51	0.52	0.47	0.52	0.49	0.55	0.48	
Reclaimed	Ore	Mt	<b>9.35</b>	0.2	1.8	0.9	0.3	0.3	1.1	0.4	0.0	1.2	1.9	1.3
	Au Grade	g/t	<b>0.619</b>	1.85	0.97	0.55	0.49	0.54	0.53	0.50	0.53	0.54	0.49	0.46
	Contained Ounces	koz	<b>186</b>	9.3	55.2	16.6	4.1	5.1	18.6	6.1	0.6	21.7	29.4	19.4
	Recovery	%	<b>88.0%</b>	93.8%	91.0%	87.6%	85.7%	85.4%	87.9%	85.1%	85.5%	87.4%	86.1%	83.0%
Processed	Ore	Mt	<b>35.3</b>	0.2	3.6	3.6	3.6	3.7	3.7	3.4	3.6	3.5	3.3	3.0
	Au Grade	g/t	<b>1.11</b>	1.83	1.37	1.27	1.34	1.34	1.11	1.13	1.07	0.89	0.68	0.74
	Total Ounces Processed	koz	<b>1,258</b>	9.4	158.9	147.5	154.7	160.5	133.9	125.2	124.8	101.6	71.3	70.6
	Total Ounces Recovered	koz	<b>1,137</b>	8.9	146.3	134.9	140.3	144.9	121.7	114.1	113.8	88.7	63.2	60.5
	Average Recovery	%	90.4	93.8	92.0	91.5	90.6	90.3	90.9	91.1	91.1	87.3	88.7	85.8
	Percentage inferred	%	0.4			0.05	0.06	0.69	0.40	0.18		2.13	1.27	

### 7.4.1 Total Material Mined

Figure 7-2 shows the mining sequence and timing graphically. As Jinkas contains 90% of the ore, it is the main focus of mining activity during the first six years of production with the high value Jackson stages 1 and 3 being slotted in years 4 and 5.

To facilitate mining of the Jackson pits, mining in Olympia stages 1 to 3 also begins in year three so that the pits can be backfilled. In year 5, when Jackson 3 enters production, the initial diversion of Warren Road becomes necessary. After mining of Jackson stage 3 is completed, the second final diversion of Warren road is completed and the pits north of the road are closed.

By the end of year 6, total material movement (TMM) capacity declines, with mining limited to Jinkas Stage 5. In year 7, mining commences in the South Zone, starting with Dingo Stage 1, and the mine life concludes with Jinkas Stage 6.

Bench turnover is shown in Figure 7-3, showing that most pits operate below the maximum limit of 16 benches (80 m) per year.

Figure 7-4 shows the monthly expit movement by stage for the pre-production period and the first two years of operations. Below that is the annual expit movement for the life of mine.

### 7.4.2 Ore Mined and Processed

The total movement of ore is presented annually in Figure 7-5 by the defined ore type as both direct feed, stockpiled and reclaimed.

Figure 7-6 presents the stockpile balance and grade by weathering after adjusting for the changes in pre-production to three months. Before year 5, the stockpile consists of ~1 Mt of low-grade material. By years 6 and 7, the stockpile increases to approximately 4 Mt, with gold grades (Au Grade) stabilizing between 0.41 and 0.53 g/t. The stockpile balance then declines towards the end of the mine life.

Figure 7-7 presents the ore feed tonnes and grade to the crusher monthly and annually reported by direct feed and reclaim, where direct feed implies that the ore is mined and processed in the same period. The strategy prioritises processing high-grade ore first, with stockpiled material supplementing plant feed in later years. The drop in feed grade after year 7 indicates a shift to lower-grade stockpiled material, a common practice to optimize overall project economics.

As the different process rates were assigned to the ore types, the schedule objective was plant operating hours rather than ore tonnes. Figure 7-8 shows the monthly and annual breakdown by feed type with the ratio of oxide and reactive transitional material in the feed.

The final charts in Figure 7-9 shows the gold production monthly and annually.

Figure 7-2 KGP Mining Schedule Mining Sequence

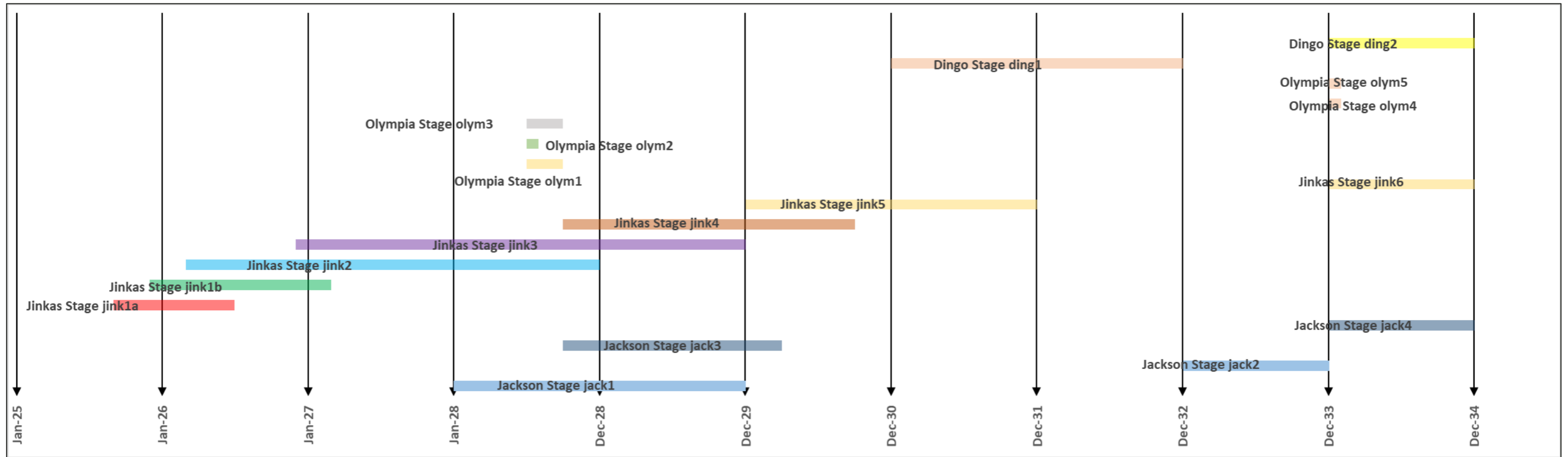


Figure 7-3 KGP Mining Schedule Bench Turnover

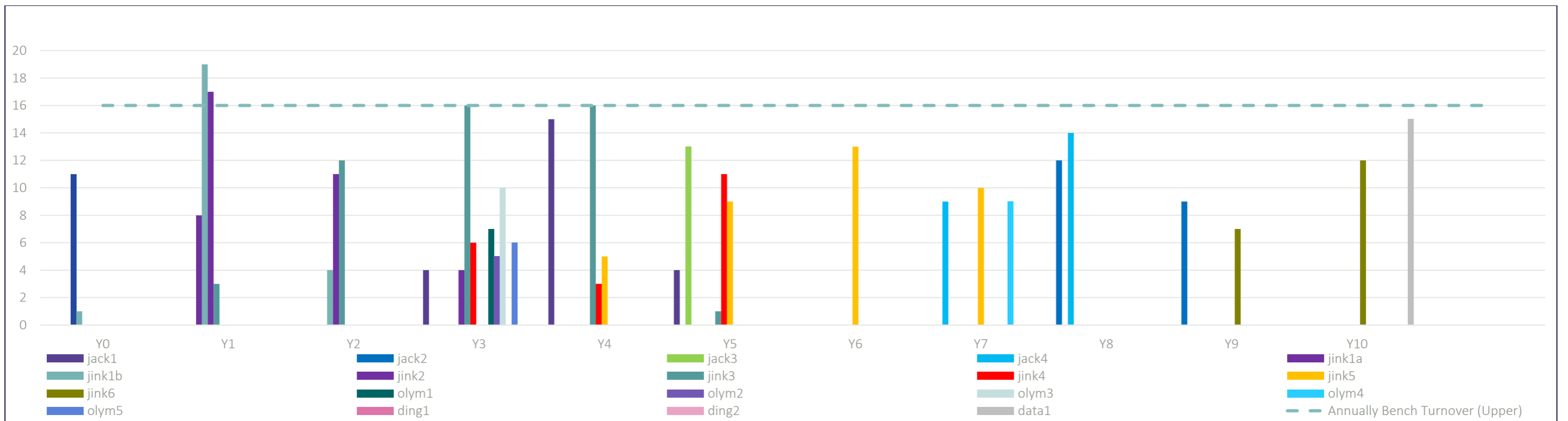


Figure 7-4: KGP Mining Schedule Ex-pit mined by Stage

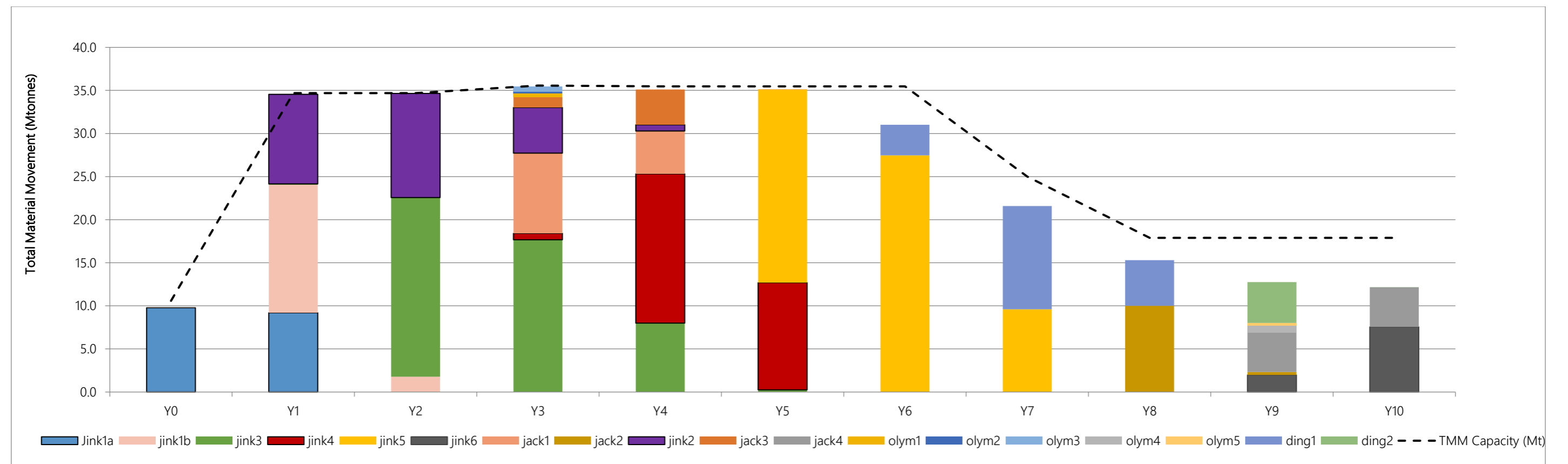
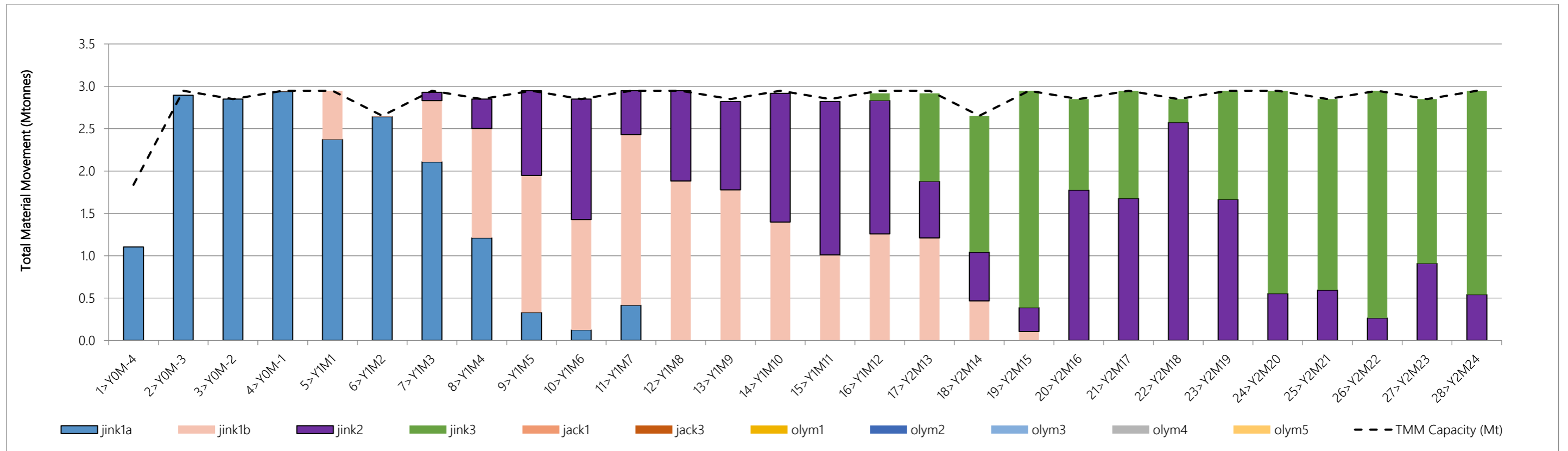


Figure 7-5 KGP Mining Schedule Ore Mining Ex-pit by Stage

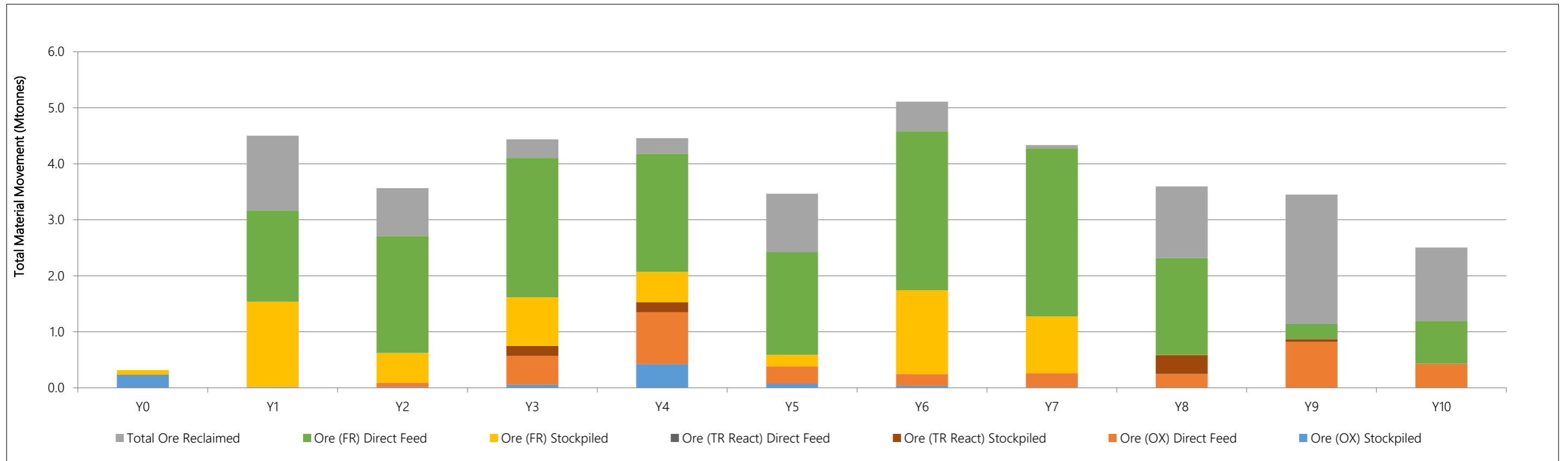


Figure 7-6 Stockpile Balance

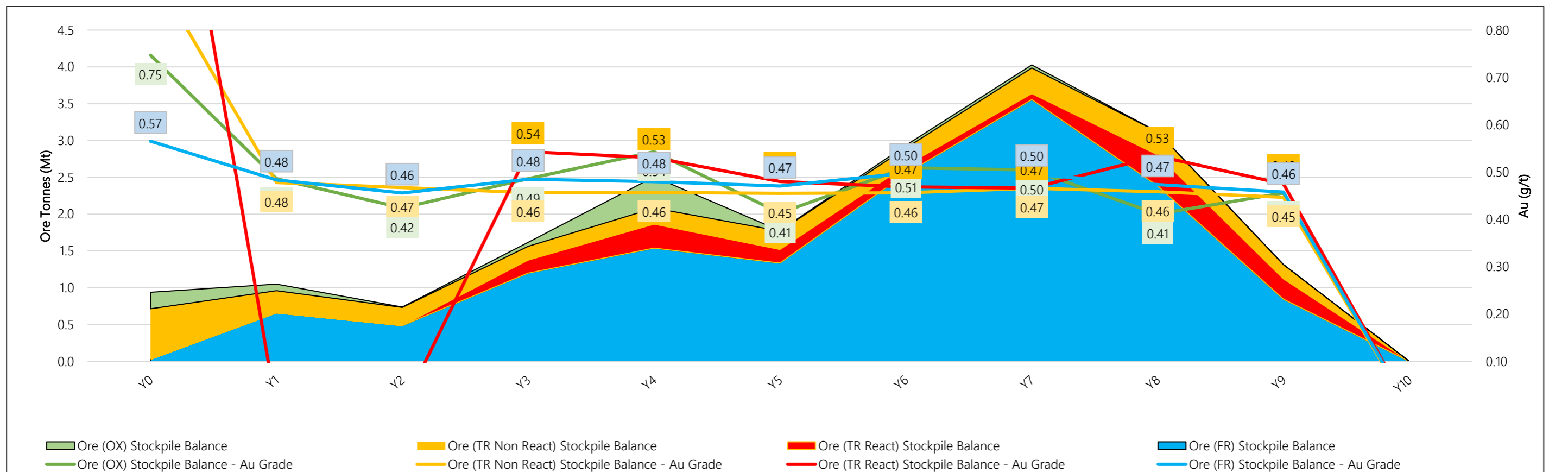


Figure 7-7 KGP Mining Schedule Crusher Feed

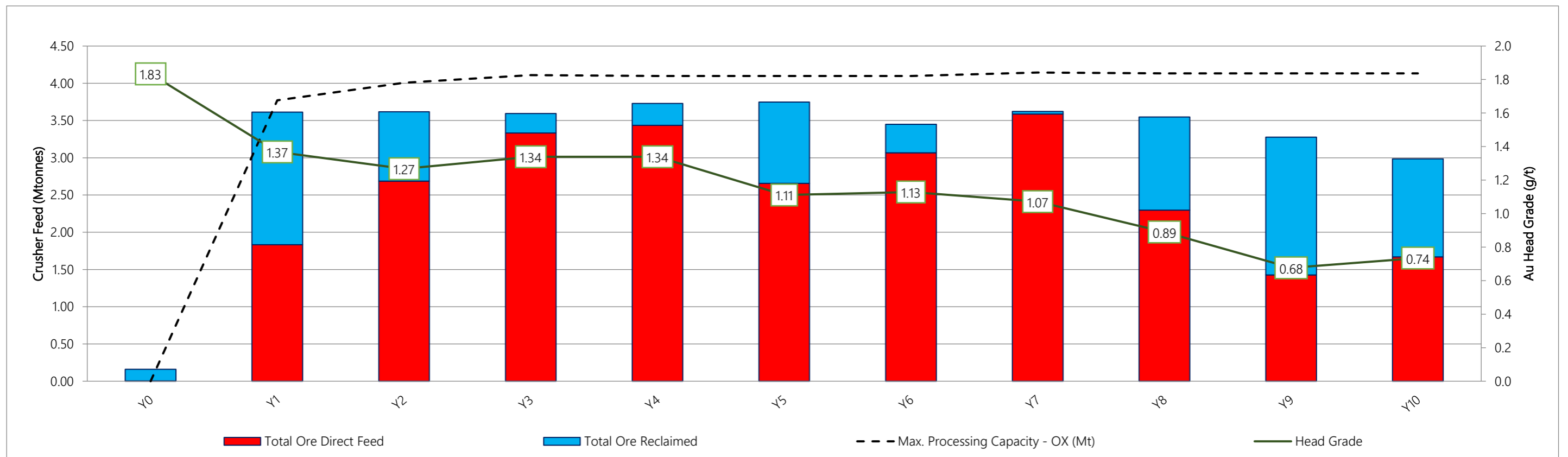
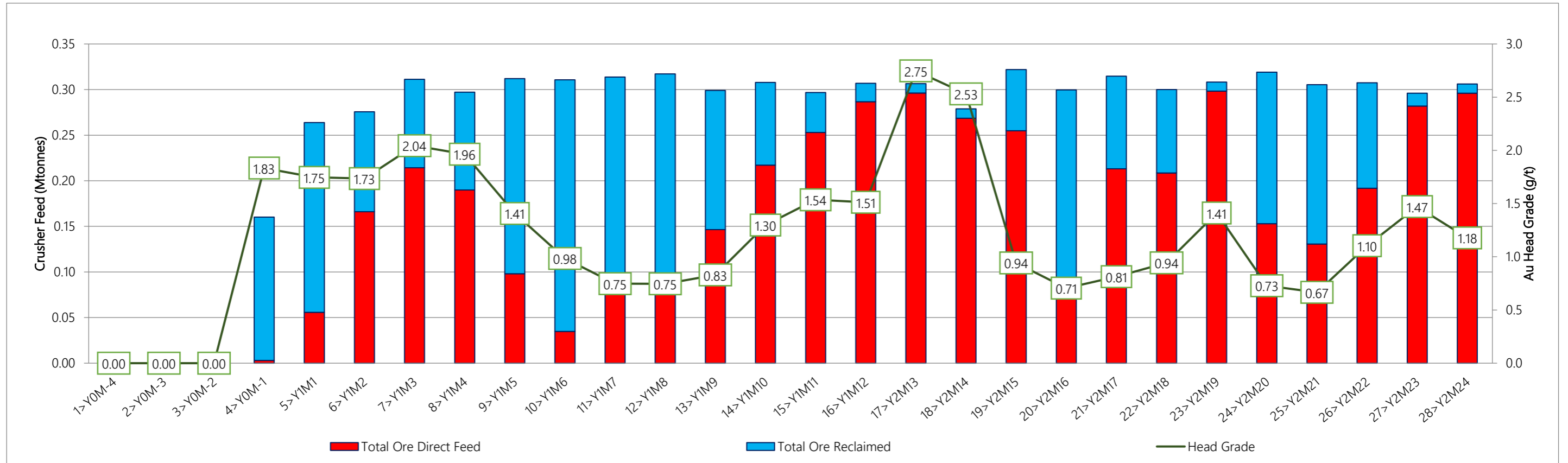


Figure 7-8 Mining Schedule Plant Feed Hours

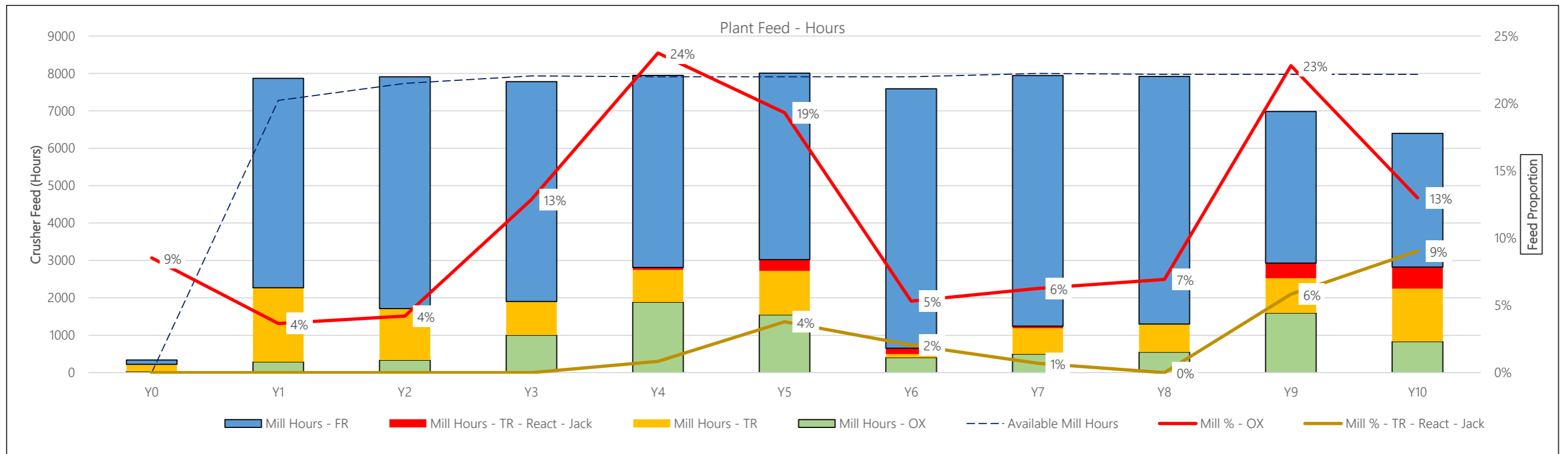
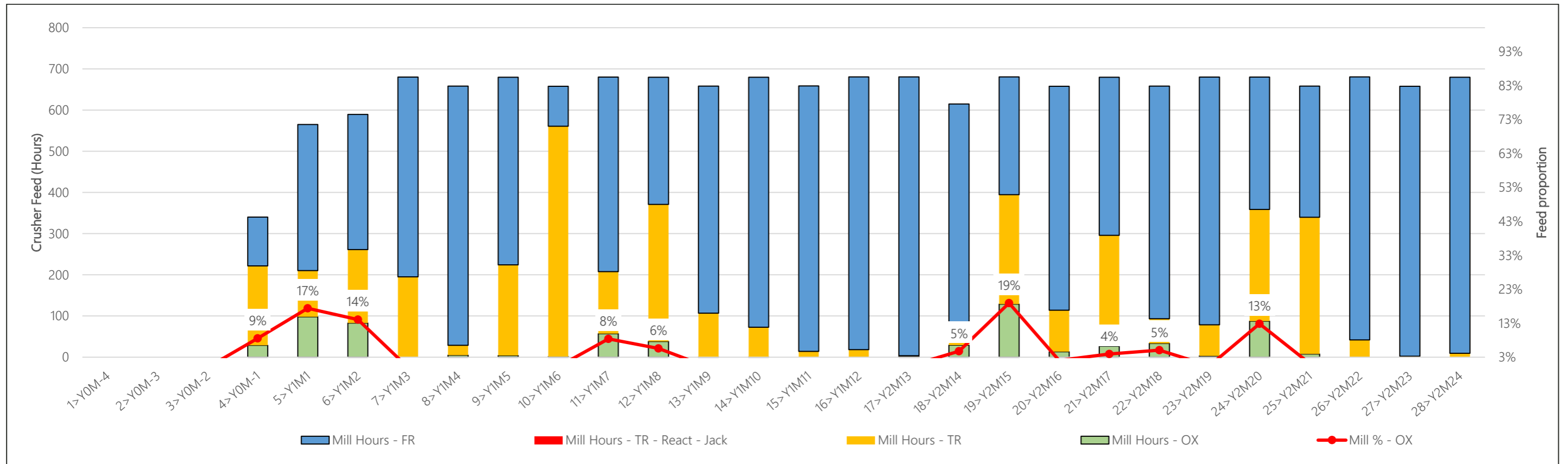


Figure 7-9 Gold Production and Recovery

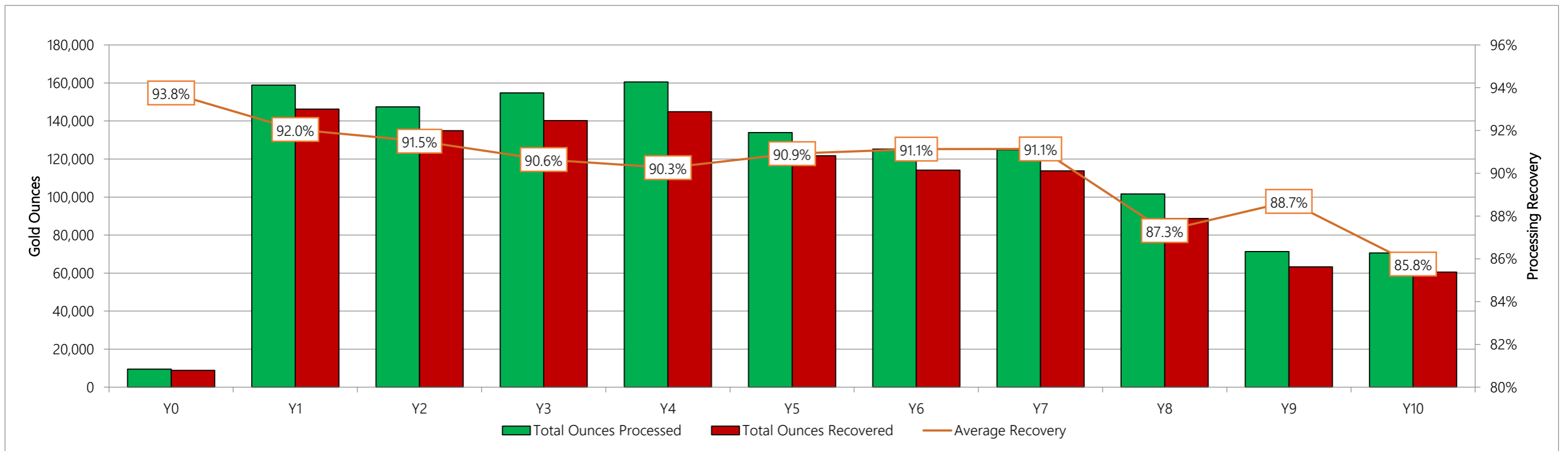
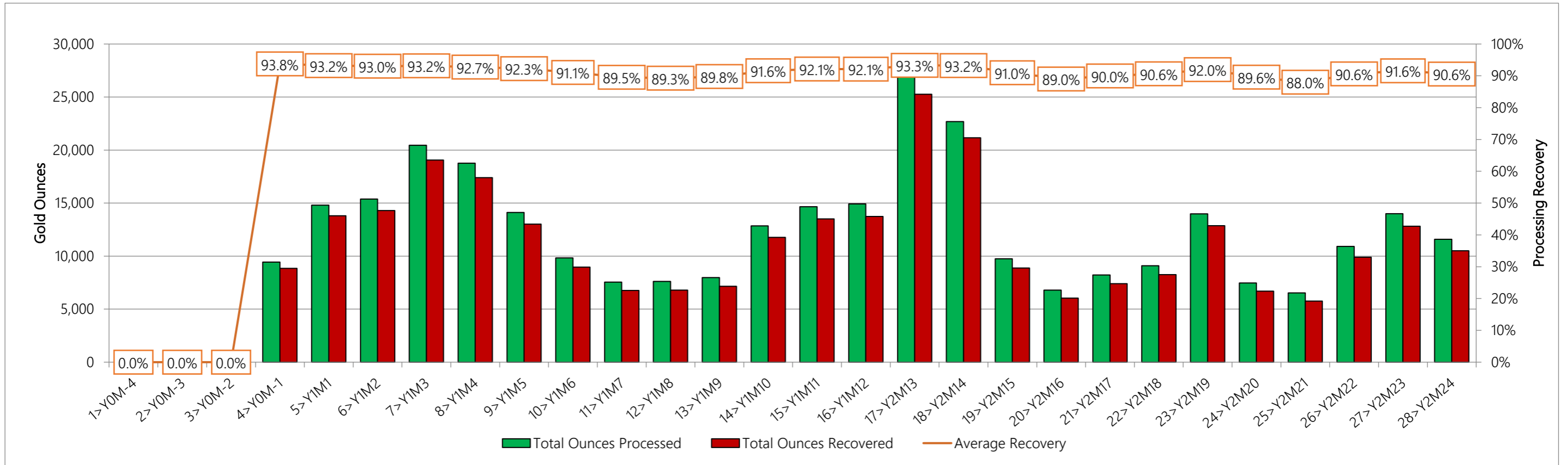


Figure 7-10: End of period plans – pre-production

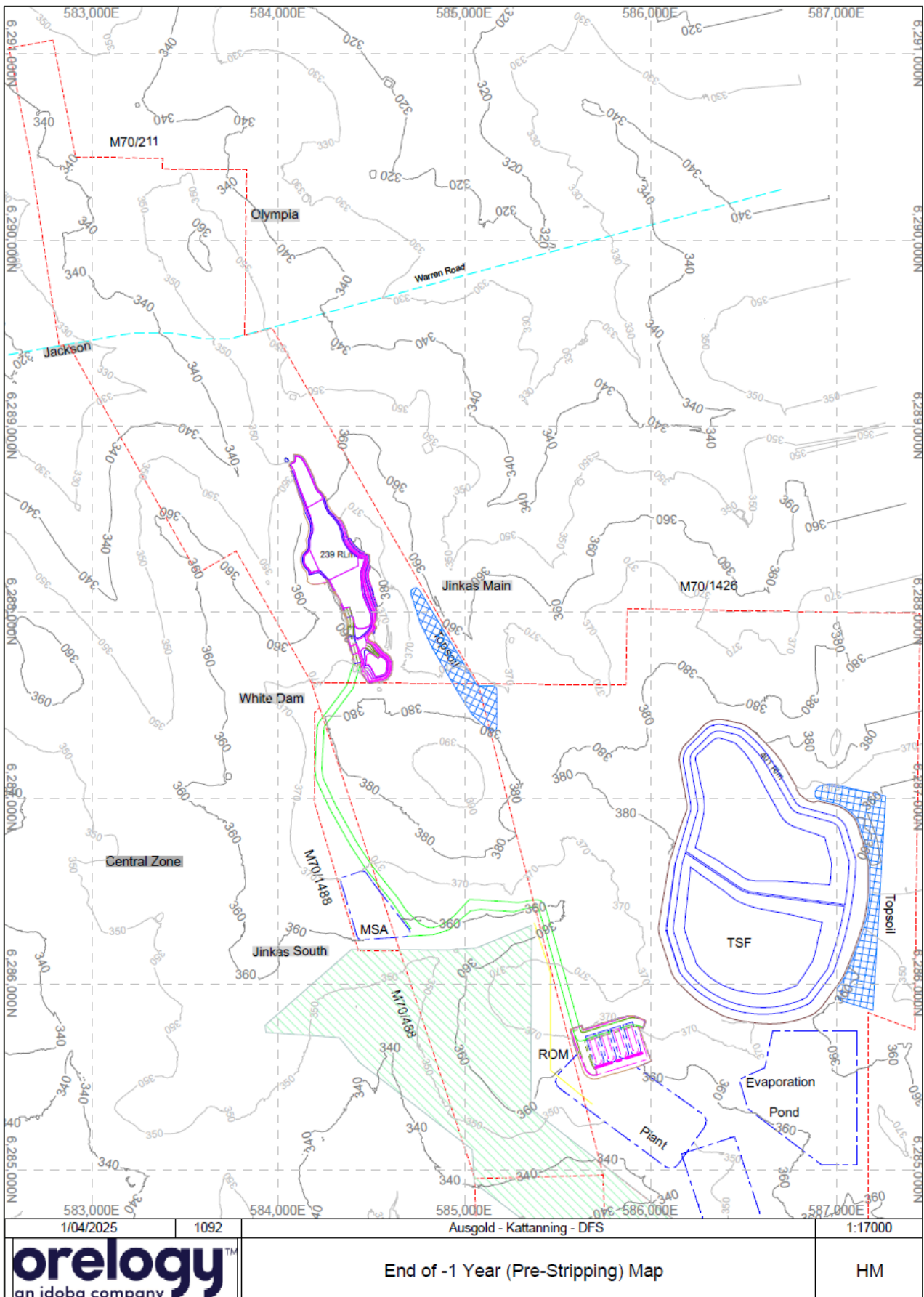


Figure 7-11: End of period plans – 1<sup>st</sup> year

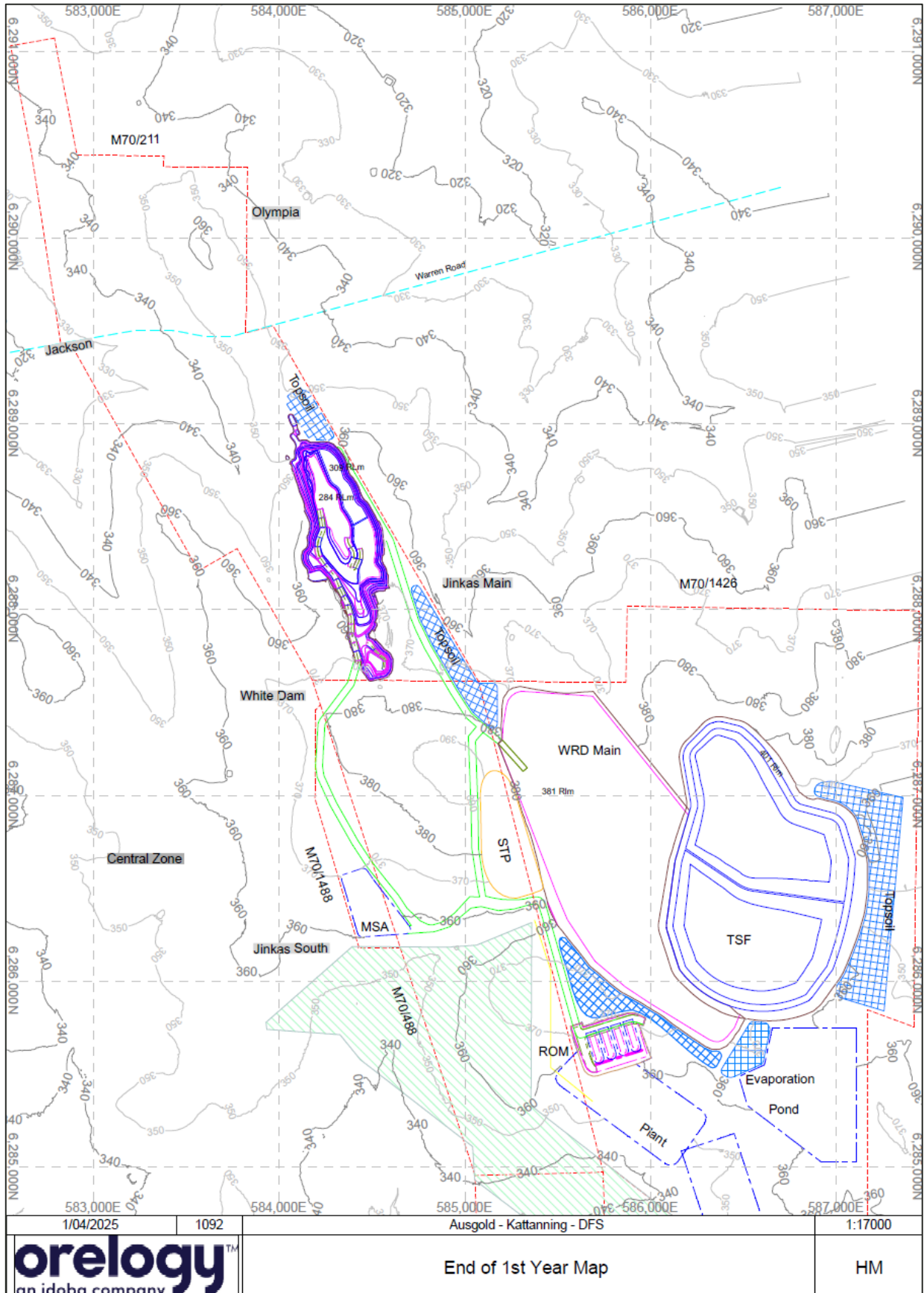


Figure 7-12: End of period plans – 2<sup>nd</sup> year

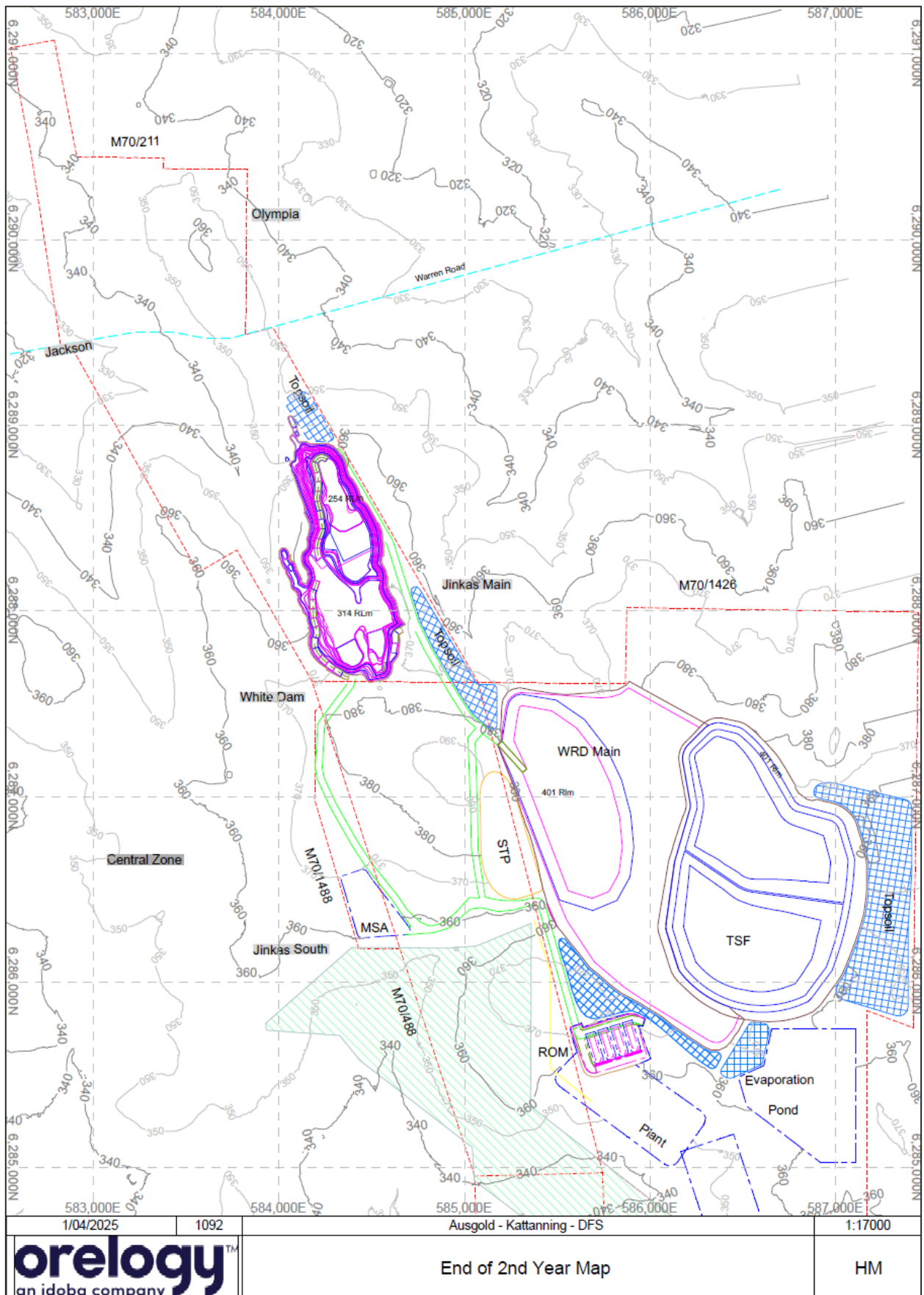


Figure 7-13: End of period plans – 3<sup>rd</sup> year

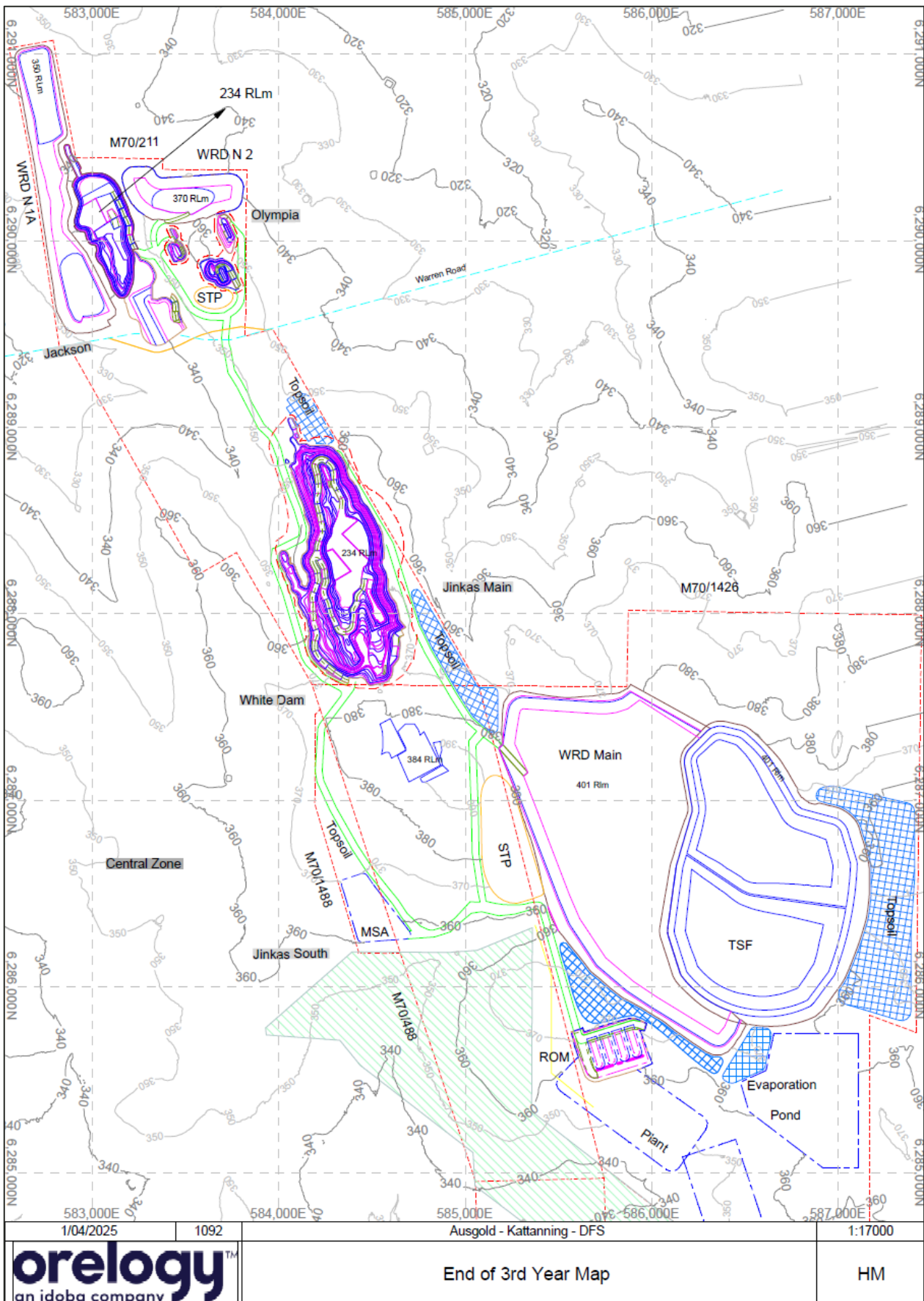




Figure 7-15: End of period plans – 5<sup>th</sup> year

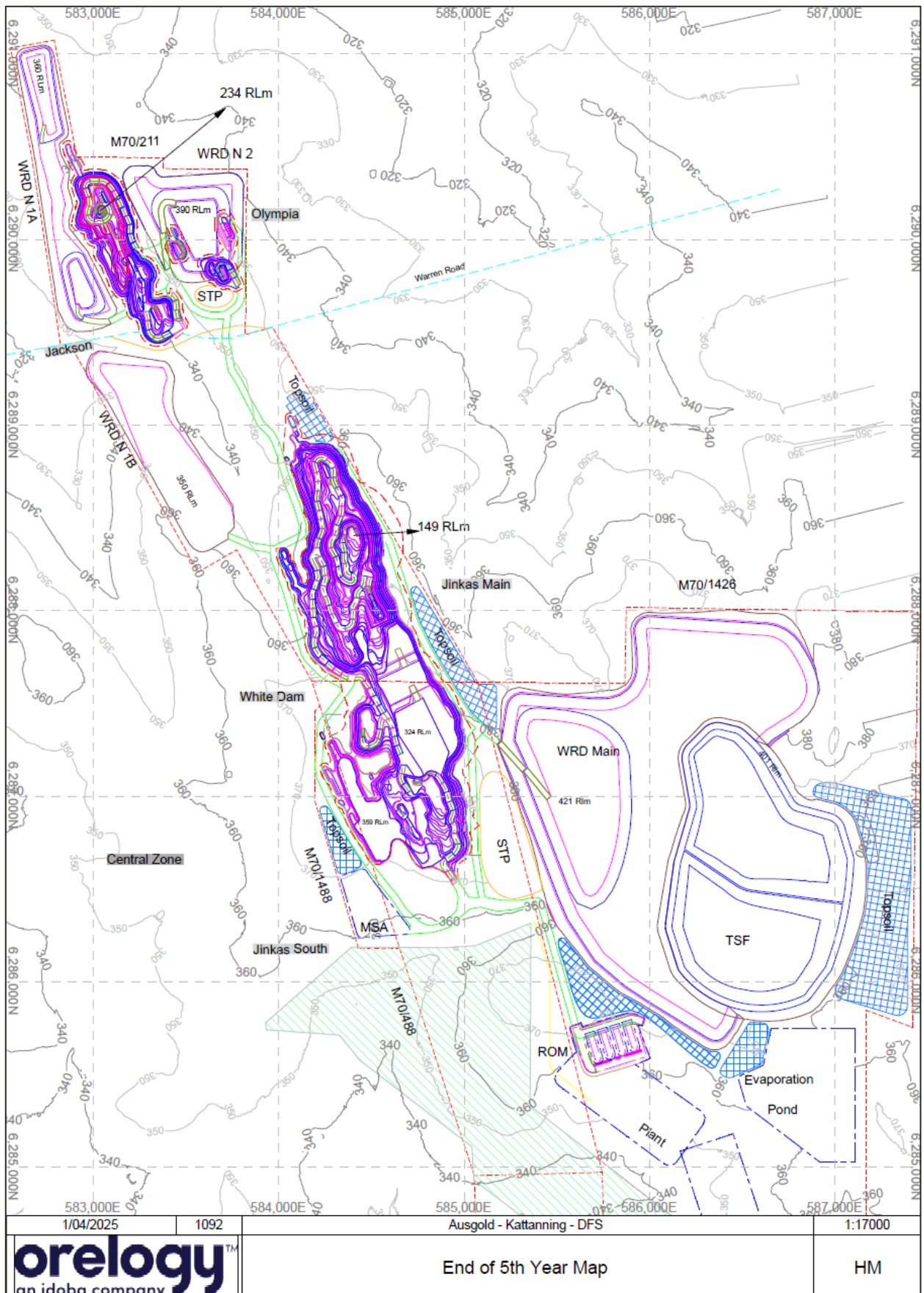


Figure 7-16: End of period plans – 6<sup>th</sup> year

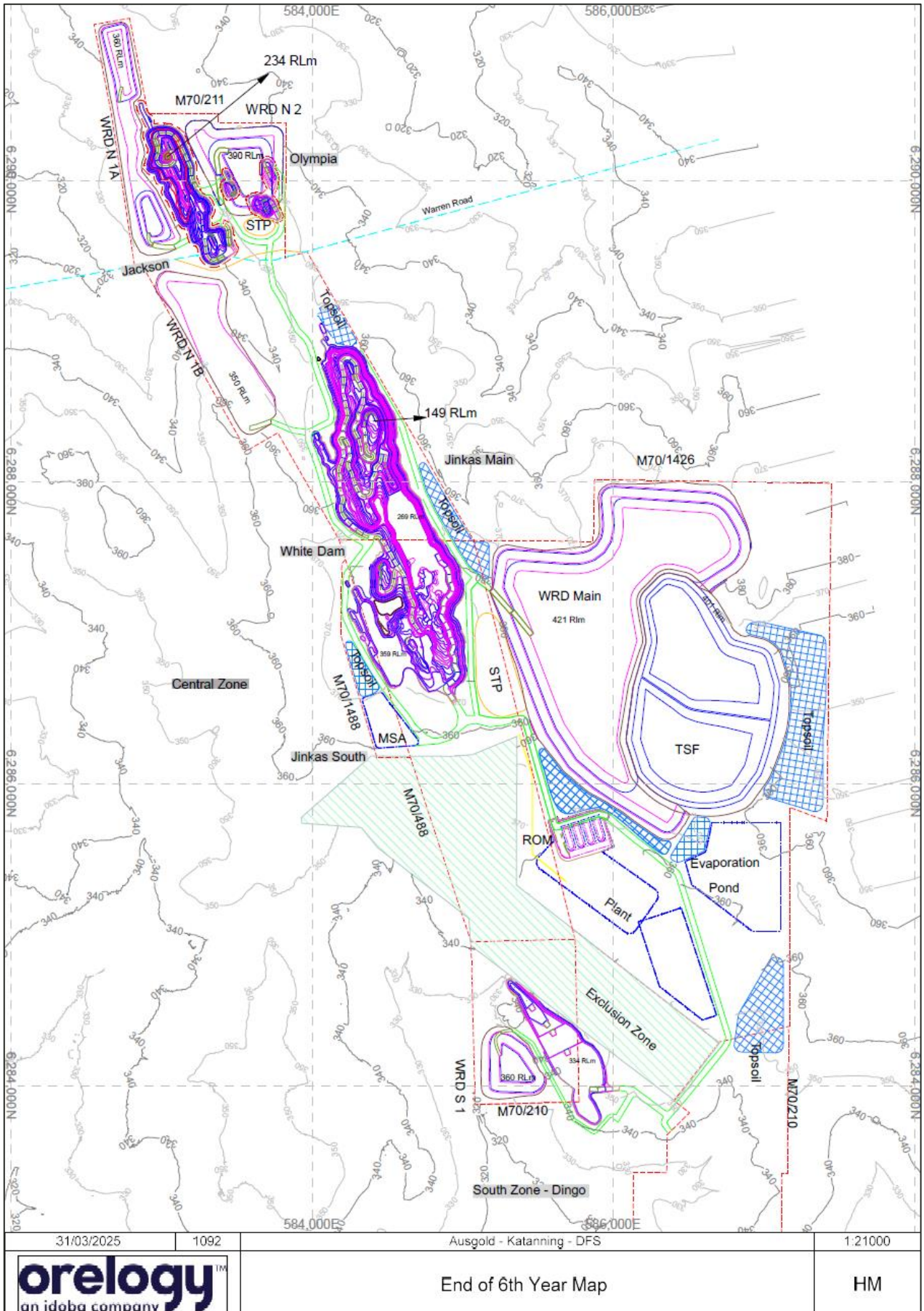


Figure 7-17: End of period plans – 7<sup>th</sup> year

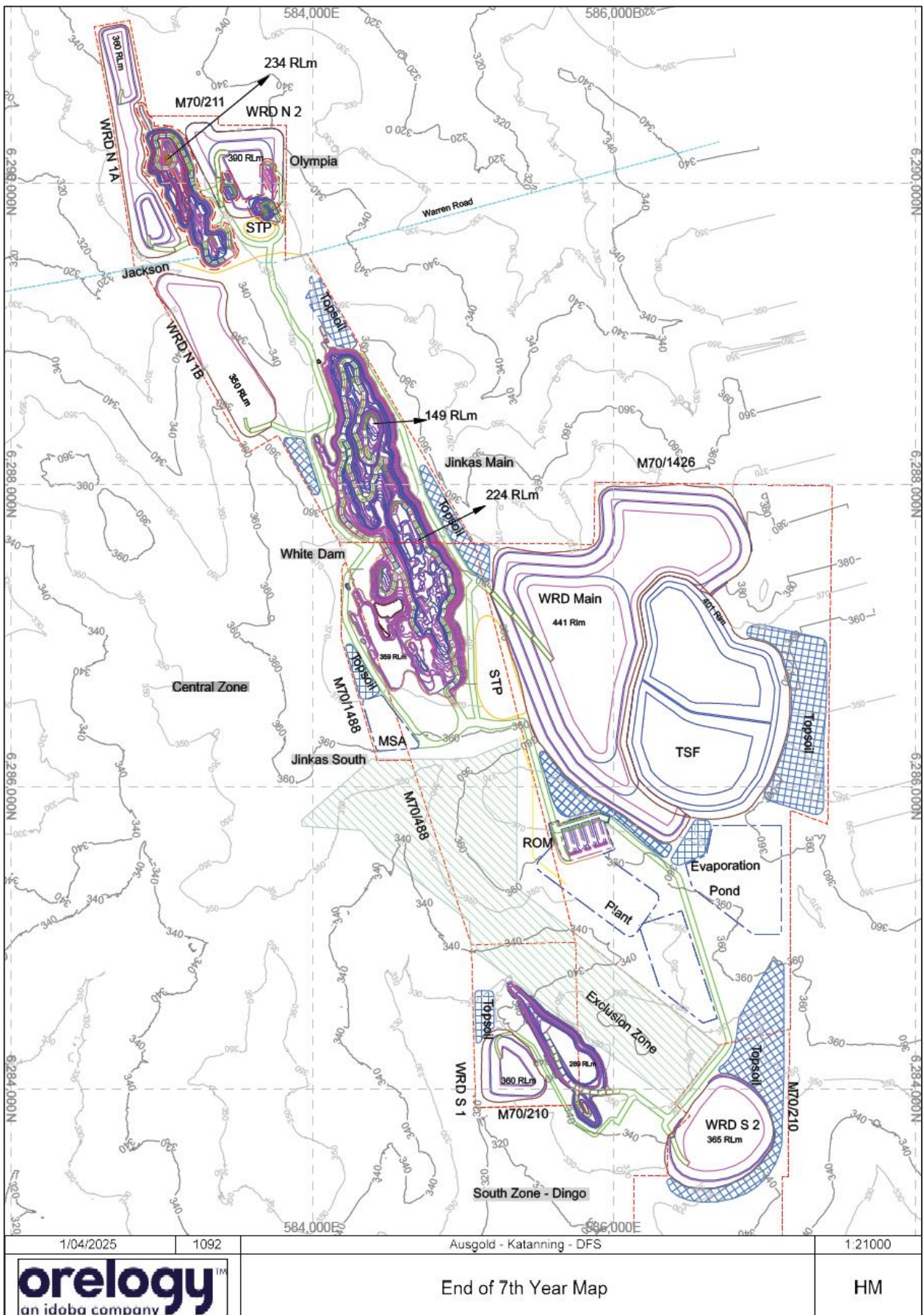


Figure 7-18: End of period plans – 8<sup>th</sup> year

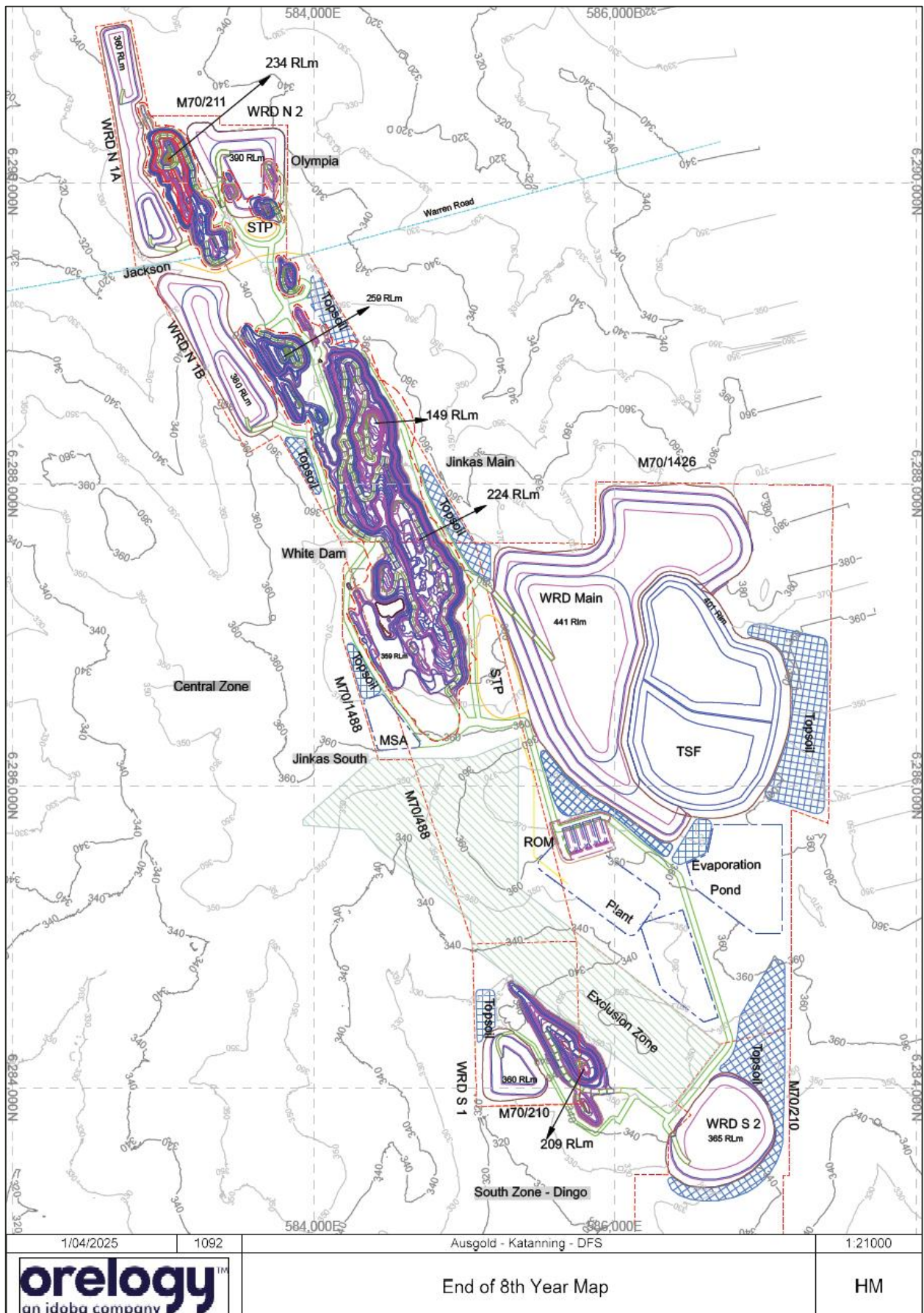


Figure 7-19: End of period plans – 9th year

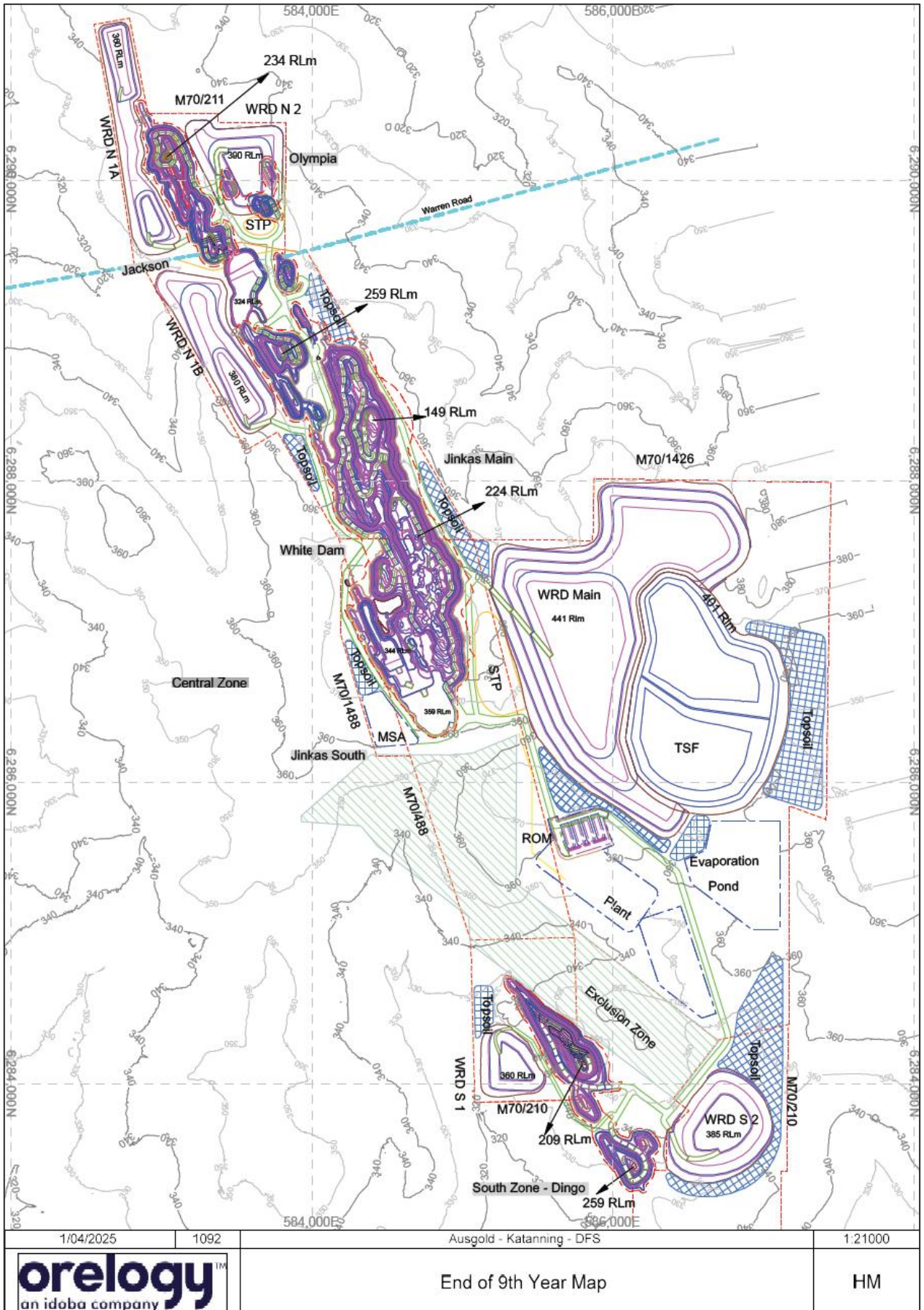
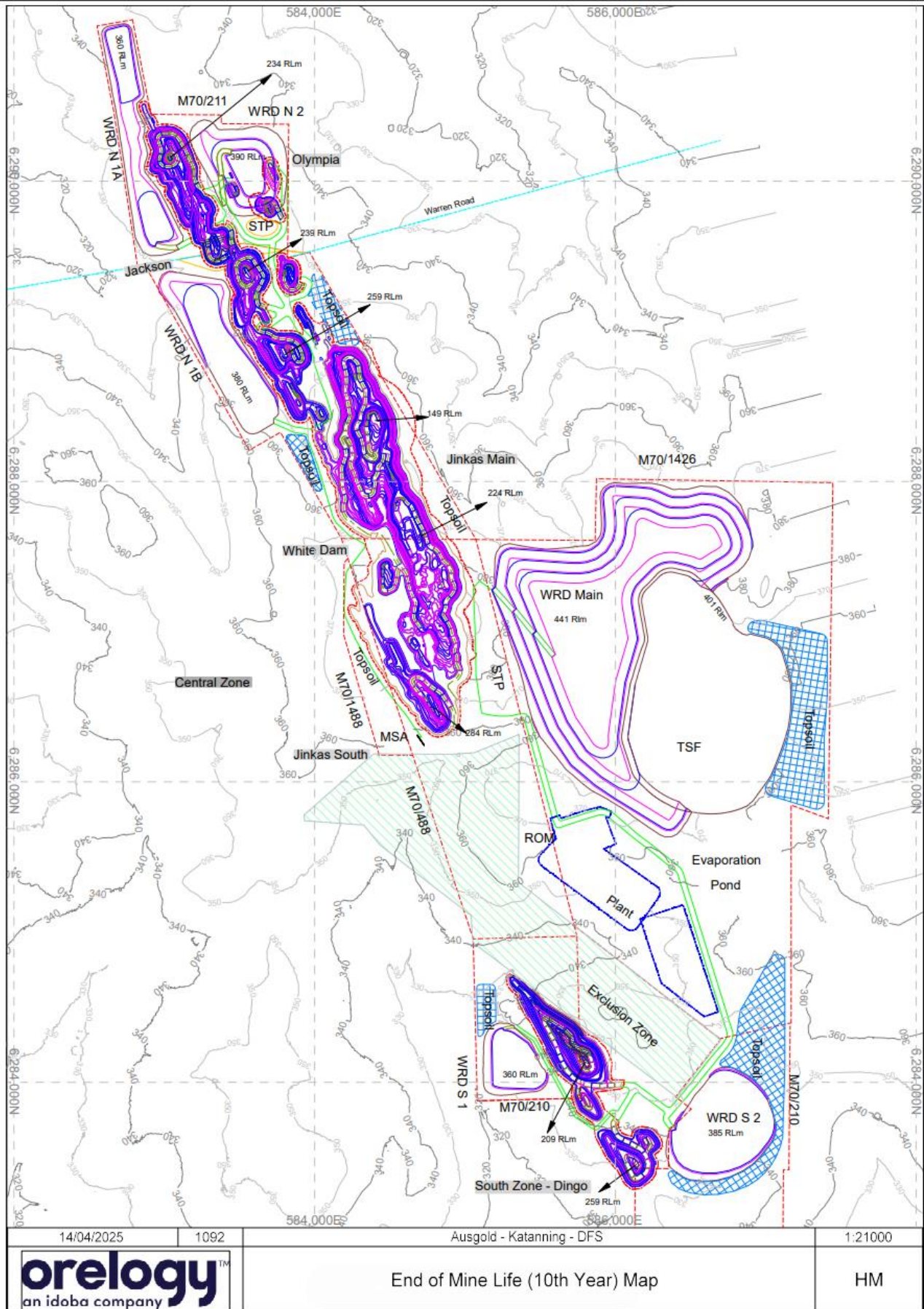


Figure 7-20: End of period plans – 10<sup>th</sup> year (Ultimate pit and waste rock dump)



## 8 MINE OPERATIONS AND MANAGEMENT

### 8.1 Mine Operations Approach

The mining operation is planned to be conducted utilising competent and experienced mining contractors. The contractor will be responsible for supply of fully maintained and operated mining equipment, supervision and infrastructure required to supply the mining services. Fuel will be supplied by Ausgold. The various aspects of the mining operation to be undertaken by the mining contractor is expected to include:

- Clearing of vegetation and topsoil stripping.
- Grade control drilling.
- Drilling and blasting.
- Pre-stripping of waste overburden.
- Mining of ore and associated waste.
- Pit dewatering up to 5 L/sec.
- Haul road construction and maintenance.
- Delivery of ore to the ROM pad or stockpiles.
- Rehandle from ROM fingers to crushers.
- Waste rock dump (WRD) rehabilitation.

Ausgold will maintain overall management for the mining operation with a mining team that will be responsible for:

- Supervision and control of the contractors (mining and grade control drilling).
- Geological, geotechnical and mine planning technical support (including survey control).
- Grade control sampling.

### 8.2 Roster

The mining operation will be continuously operated 7 days per week. The mining shifts will commence at 6am and finish at 6pm and include time for meetings at the start of each shift, pre-start checks, and handover at the end of each shift. There will be several rosters utilised at the site. These consist of:

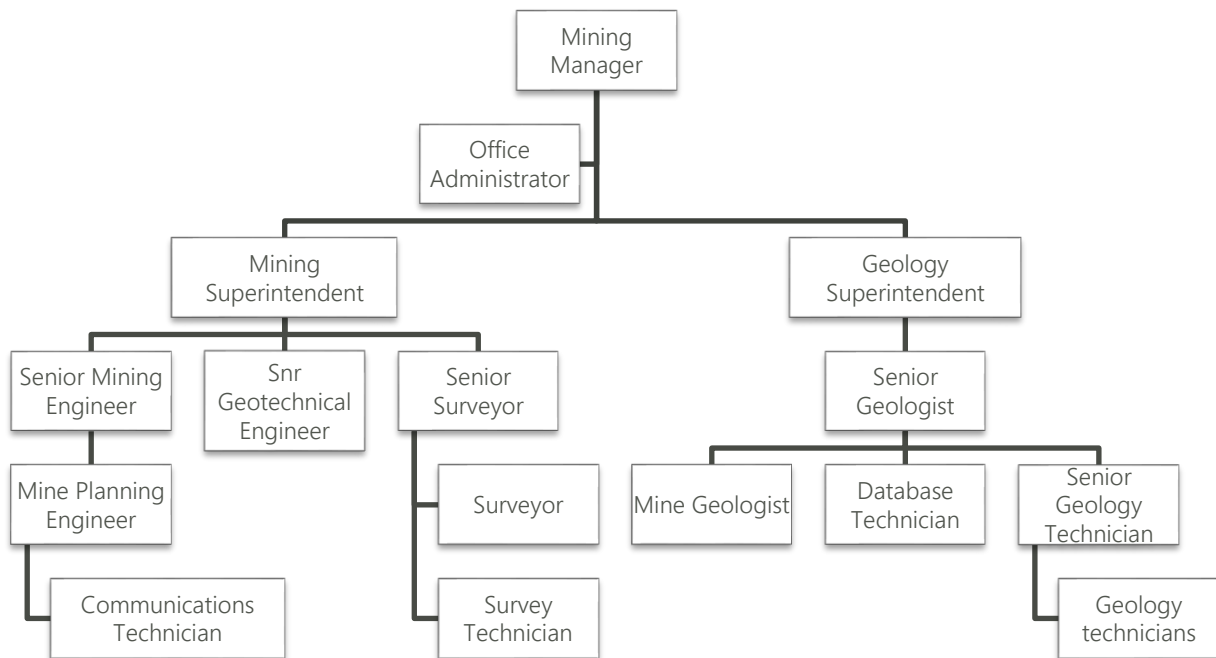
- Roster 1 – 5 days on and 2 days off. Used for senior personnel.
- Roster 2 – 8 days on and 6 days off. Day shift only. This will be used for technical and service staff for positions that do not require night shift cover.

- Roster 3 – 7 day shifts, 7 night shifts, and 7 days off. This will be used for all personnel required to operate or maintain the mining equipment.

### 8.3 Mine Management & Technical Services

The structure of the Owners mining team is presented in Figure 8-1. The Mining Manager will be responsible for safe operation of the site as the Senior Site Executive (SSE). In the absence of the Mining Manager, the Process Manager will take on the role of SSE and the Mining Superintendent will take on regulatory responsibilities as Quarry Manager.

Figure 8-1 Organisational Structure for the Owners Mining Team



### 8.4 Manning Levels

While close to the towns of Katanning, the site is expected to operate on a fly-in fly-out or drive-in drive-out basis. Ausgold plan to establish a camp at the Katanning to support the 24 hours per day operating regime. The camp will be of sufficient size to cater for the Mining Contractors personnel with accommodation and meals provided.

The Owner’s team will total 29 personnel for the first 6 years, reducing to 25 personnel for three years and finally 17 personnel for the final two years.

Contractor personnel numbers will vary between 192 to 201 over the first 10 years before reducing to between 155 to 135 after demobilising the third fleet.

## 9 INFRASTRUCTURE AND SERVICES

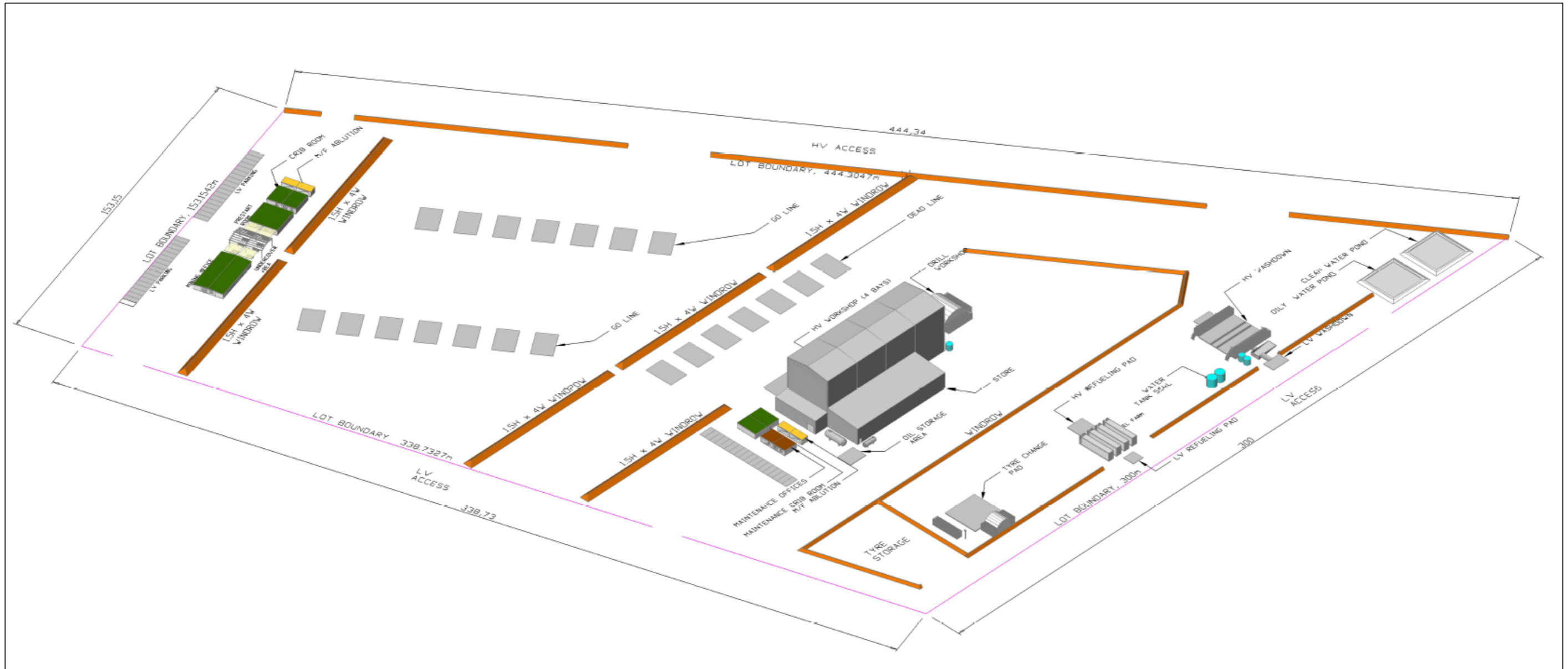
### 9.1 Mine Services Area

The mine services Area (MSA) was designed by a Contractor and includes:

- A mining office with prestart/training room, crib room and ablutions.
- A 4 bay heavy vehicle dome workshop and an adjacent deadline for up to seven trucks.
- A maintenance office, crib room and ablutions adjacent to the HV workshop.
- A drill workshop comprised of two containers connected by a dome cover.
- An enclosed warehouse 45 m by 15 m.
- A tyre change workshop and adjacent tyre storage area
- Bulk lubes facility.
- Fuel facility comprised of a pad, pumping and dispensing system.
- Separate heavy vehicle and light vehicle washdown pads with oily water catchment and treatment facility.
- Black water collection system.
- An open area approximately 150 m by 150 m for the go-line.

A 3d rendering of the MSA is presented in Figure 9-1.

Figure 9-1 Mine Service Area Layout



## 9.2 Site Preparation

### 9.2.1 Pre-Production Dewatering

Dewatering bores planned by Rockwater Consulting will be installed prior to commencement of mining operations. Water from these bores will be directed to the raw water dam at the plant.

A small amount of water in the existing pits will also be pumped to the raw water dam prior to mining.

### 9.2.2 Haul Roads

Road construction comprises development of 6.4 km of haul roads from the site access to the MSA through to the ROM pad and to the Jinkas Stage 1a pit. A temporary haul road across the Main WRD to the TSF embankment area will also be required.

A total of 15.4 km of haul road will require construction over the life of the operation.

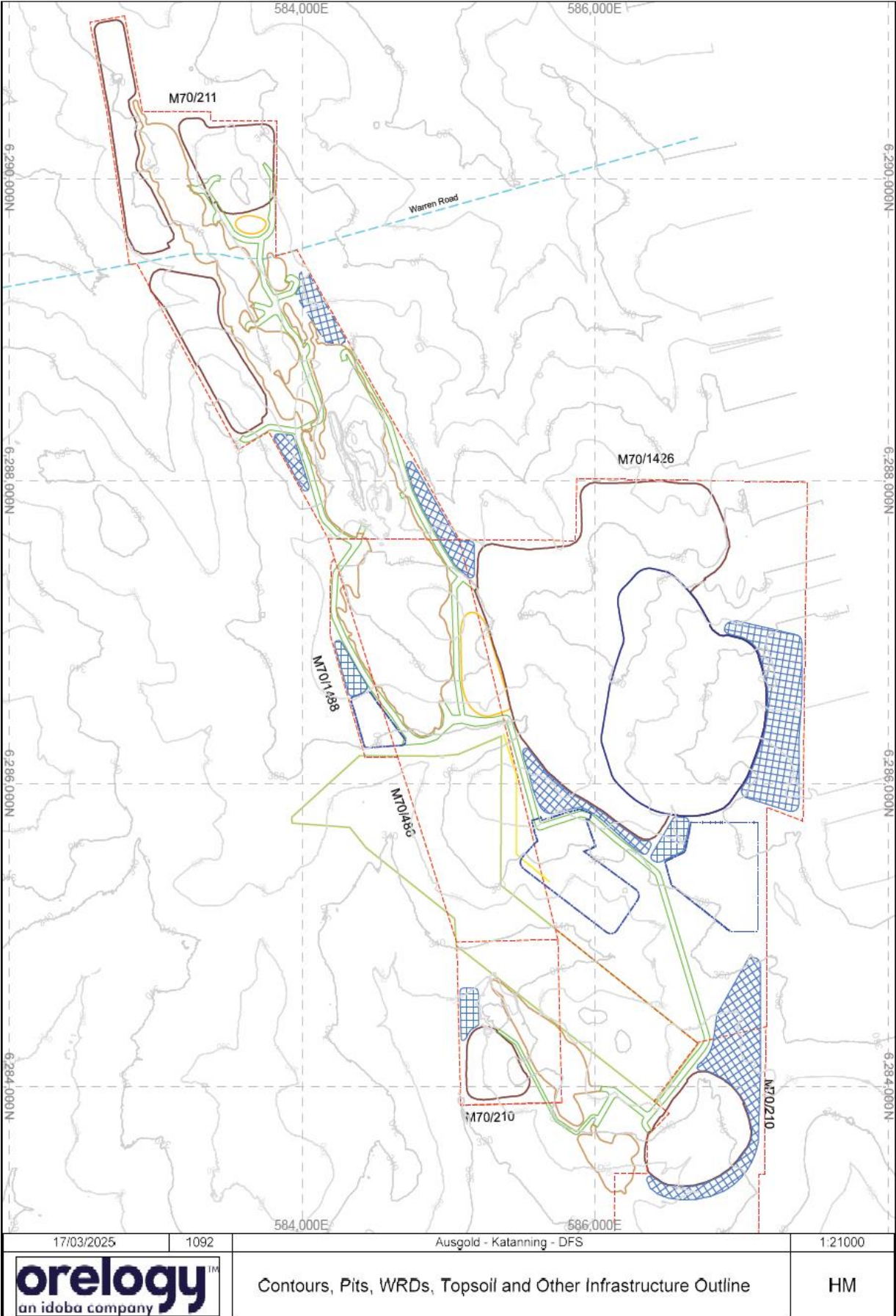
### 9.2.3 Clear and Grub

Table 9-1 outlines the clearing areas and estimated volumes of topsoil for the site, excluding the plant area and initial area for the southern TSF starter dam undertaken by the civil earthworks contractor. All scrub and timber will be cleared prior to topsoil removal. Topsoil will be hauled to stockpiles around the periphery of the waste dump for future rehabilitation as shown in Figure 9-2.

**Table 9-1: Clearing Area & Topsoil Volumes**

Outline	Clearing (Ha)	Topsoil (BCM)
TSF	77	231,900
Roads	56	166,500
WRD main	198	593,838
Jinkas pit	126	378,352
Jackson Pit	50	150,048
Olympia Pits	9	26,152
WRD North1 (north of Warren Rd)	61	183,000
WRD North1 (south of Warren Rd)	36	106,723
STP MSA	36	106,605
Dingo Pit	33	98,496
WRD Dingo	55	163,735
<b>Total</b>	<b>735</b>	<b>2,205,348</b>

Figure 9-2: Pit and Waste Dump Extents (together with area needed for clearing)



## 10 MINING COST ESTIMATES

### 10.1 Basis of Estimate

The mining cost estimate for the Project assumes that the Ausgold engage a suitably qualified contractor for the mining services including mine production, maintenance and supervisory functions, with technical services and management functions performed by an inhouse Ausgold team.

The costs are based on the physicals derived from each of the mine plans described in Section 8 and then developed from a combination of the physicals together with mining contract rates. The primary sources were:

- Mining Contractor budget price submissions from Mar 2025.
- A fuel price of \$1.10 per litre.

The overall cost estimate is considered to have an accuracy of +/- 10%.

All costs/prices are provided in Australian dollars.

## 10.2 Contractor Submissions

A request for Budget Pricing was sent to a total of 7 contractors, of which 4 responded. The quantities in the RFBP were preliminary and may not match final quantities in the mine plan.

All submissions conformed to the RFBP with three of the four very close in overall pricing as presented in Table 10-1. Excluding Contractor 1, the closeness in overall pricing provides confidence in the estimates.

Table 10-1 Contractor Submissions

	Description/Activity	Unit	Contractor 1	Contractor 2	Contractor 3	Contractor 4
1	Mobilisation	\$M	5.4	4.3	4.1	3.0
2	Establishment	\$M	44.1	21.4	7.0	2.9
3	Demobilisation	\$M	4.1	3.1	4.0	1.3
4	Dis-Establishment	\$M	0	1.6	0.5	0.9
5	Monthly Management Fees	\$M	332.6	209.3	265.3	143.3
6	Drill & Blast	\$M	334.5	286.5	210.1	219.8
7	E,L,H & D - Ore	\$M	173.2	120.3	84.8	100.4
8	E,L,H & D - Waste	\$M	514.3	528.3	610.6	634.3
9	ROM Management Mill Feed	\$M	20.3	47.1	28.9	43.7
10	Clearing And Grubbing	\$M	1.7	2.1	1.7	2.5
11	Topsoil Management	\$M	14.0	10.8	10.4	15.5
12	Road Construction	\$M	10.8	1.8	1.0	2.7
13	Pit Dewatering	\$M	6.0	4.0	21.7	2.1
14	Fuel	\$M	192.7	206.9	179.0	206.5
	<b>Estimated Total Contract Value</b>	\$M	<b>1,653.8</b>	<b>1,447.4</b>	<b>1,429.1</b>	<b>1378.9</b>
	<b>Total Unit Cost Mined</b>	<b>\$/t</b>	<b>5.96</b>	<b>5.22</b>	<b>5.15</b>	<b>4.97</b>

Analysis of submissions:

- The submission from Contractor 1 was significantly high than the other three in almost all areas, with the exception of waste haulage and fuel which could be attributed to the selection of a larger 180 t dump truck over the 140 t truck suggested in the RFBP.
- Contractors 3 and 4 provided low establishment costs with no breakdown to support the costs, whereas Contractor 2 provided full details of the site establishment costs with detailed plans. The site establishment costs provided by Contractor 1 were considered excessive.
- The mobilisation costs from Contractors 2 and 3 were mid-range compared to the other submissions. Further detailed review identified that Contractor 3 had neglected to include mobilisation costs for the two front-end loaders.

- The direct mining costs for Contractors 2 and 3 were mid-range at \$4.85/t. This was within 5% of the mining cost used in pit optimisation. The direct mining costs for Contractor 4 at \$4.64/t was within 1% of the mining cost used in pit optimisation.
- Monthly management fees ranged from 20% to less than 10% of the contract cost.
- Contractors 2 and 4 had the highest fuel consumption which can be attributed to the equipment selection of 250 t excavators matched to 140 t trucks compared to 350 t excavators and 180 t trucks selected by Contractors 1 and 3.
- Drill and blast costs for Contractors 3 and 4 were at the lower range of expectations.
- ROM management costs from Contractor 1 and 3 appear to be too low at less than \$0.85/t.
- Pit dewatering costs provided by Contractor 3 were considered out of range in comparison to the other three submissions.

Contractor 4 was selected as the preferred option for the following reasons:

- The submission was the lowest and aligned closely with the costs used in the pit optimisation.
- Site Establishment costs were considered to be appropriate for a low capital startup operation.
- At 15% of the contract cost, the fees from Contractor 4 were considered to be the correct balance of risk and incentive to meet production targets.

## 10.3 Capital Costs

Capital Costs for the project total \$43.3 M comprised of:

- Owner Equipment - \$0.61 M
- Contractor Site Establishment - \$2.90 M
- Contractor Mobilisation - \$2.87 M
- Pre-production mining cost - \$36.89 M

Details of the capital costs are provided in the following sections.

### 10.3.1 Owners Equipment

Table 10-2 provides a breakdown of the capital costs for survey equipment and mine planning software for the Owners technical services team.

**Table 10-2 Owner Equipment**

	Description	Amount (\$)
Survey Equipment	Total Station	101,238
	RTK GPS Base Station	31,496
	Rover with Controller	56,243
	Drone	22,497
	Drone software (PIX 4D)	16,873
Mine Planning Software	Vulcan Surveyor	33,116
	Drill and Blast	13,093
	Mine Design Upgrade OC	24,953
	Vulcan GeoStat Modeller	83,484
	Grade Control	41,588
	Vulcan Mine Modeller OC	74,705
	Maptek Evolution Scheduler	110,902
<b>TOTAL</b>		<b>610,188</b>

### 10.3.2 Contractor Establishment

Table 10-3 provides a breakdown of the site establishment costs provided by Contractor 2.

**Table 10-3 Contractor Site Establishment**

Description	Total Amount (\$)
Preliminaries, trenching, electricals and lighting	1,139,322
Contractor's Workshop - 4 Bays	700,983
Contractor's Drill Workshop - Dome and Containers	57,724
Warehouse / Stores and Fencing Area	22,815
Fuel Facility - pad, pumping and dispensing system	82,263
Bulk Lube Facility	136,505
Wash Down Facility	171,794
Oily Water Collection and Treatment Facility	46,776
Tyre Bay	56,944
Office, crib, ablutions, pre-start and maintenance office	298,162
Black water collection system, potable water reticulation and tank	26,772
Light and Heavy Vehicle Parking Areas	15,901
Communication reticulation, radios and radio system	74,135
Turkeys Nest & Earthworks	31,027
Explosives Storage (Bulk and Initiation Compounds)	38,879
<b>TOTAL</b>	<b>2,900,002</b>

### 10.3.3 Mobilisation

Table 10-4 provides a breakdown of the mobilisation costs provided by Contractor 2.

Table 10-4 Contractor Equipment Mobilisation

Description	Unit	Qty	Unit Price (\$)	Total Amount (\$)
Mobilise Contractor Personnel - Operations	ea	213	1,800	363,400
Hitachi EX2600	ea	3	108,000	324,000
Hitachi EX1900	ea	1	81,000	81,000
Cat 992	ea	1	108,000	108,000
Cat 988	ea	1	81,000	81,000
Cat 785D	ea	24	54,000	1,296,000
Cat 777G	ea	4	32,400	129,600
Cat D10T	ea	4	54,000	216,000
Cat 16M	ea	2	21,600	43,200
Cat 777w/c	ea	2	32,400	64,800
336DL	LS	1	16,200	16,200
Lighting Plants	LS	16	1,080	17,280
Light Vehicles	LS	11	1,080	11,880
Light Trucks	LS	2	3,240	6,480
Bus	LS	2	3,240	6,480
Service Truck	ea	1	3,240	3,240
Fuel Truck	ea	1	3,240	3,240
Mobile Crane	ea	1	3,240	3,240
IT Loaders	ea	1	3,240	3,240
Forklift/Telehandler	ea	2	3,240	6,480
Tyre Handler	ea	1	3,240	3,240
Drill & Blast Fleet	ea	8	8,085	64,680
<b>TOTAL</b>				<b>\$2,872,680</b>

### 10.3.4 Pre-Production Costs

Table 10-5 provides a breakdown of the pre-production direct mining costs.

Table 10-5 Pre-Production Costs

Cost Centre	Unit	Y0 M-4	Y0 M-3	Y0 M-2	Total
Clear & Topsoil	\$	424,079	424,079	424,079	1,272,237
Road Construction	\$	271,548	271,548	271,548	814,644
L&H Waste	\$	3,792,870	7,919,453	7,026,209	18,738,532
L&H Ore	\$	27,248	601,314	722,659	1,351,221
D&B Waste	\$	627,326	1,488,787	1,325,586	3,441,699
D&B Ore	\$	5,551	127,666	136,277	269,494
Fuel Consumption	\$	1,837,344	1,856,546	1,882,026	5,575,916
Contractor Overheads	\$	1,168,525	1,168,525	1,168,525	3,505,575
Owner Overheads	\$	372,559	372,559	392,465	1,137,583
Grade Control	\$	21,314	463,999	250,775	736,088
Pit Dewatering	\$	17,296	17,296	17,296	51,888
<b>TOTAL</b>	<b>\$</b>	<b>8,565,660</b>	<b>14,711,772</b>	<b>13,617,445</b>	<b>36,894,877</b>
Total Mined	Tonnes	1,104,230	2,894,584	2,849,153	6,847,967
Unit rate	\$/t	7.76	5.08	4.78	5.39

## 10.4 Operating Costs

### 10.4.1 Owner Costs

The mining personnel employed by Ausgold will be required to oversee and manage the day to day activities and the contractors. Leave will be covered within the team. On-costs of 31% are included for superannuation (11.5%), payroll tax (5.5%), workers compensation (1.6%), ongoing recruitment and training allowance of 3.8% and production bonus of 5%. Personnel requirements are shown in Table 8 2. The table includes counts for computers and light vehicles.

Table 10-6 Owners Minig Department Personnel

Position	Roster	Gross per Person (\$/year)	Loading Units		Computer	Light Vehicles	TOTAL (\$)
			2	3			
Mining Manager	5 on/2 off	317,778	1	1	Y	Y	3,177,780
Tech Services Superintendent	8 on/6 off	271,604	1	1	Y	Y	2,716,040
Senior Mining Engineer	8 on/6 off	220,342	1	1	Y	Y	2,203,420
D&B Engineer	8 on/6 off	172,122	1	1			1,721,220
Mining Planner	8 on/6 off	215,720	1	1	Y		2,157,200
Senior Geologist	8 on/6 off	210,681	1	1	Y	Y	2,106,810
Geologist	8 on/6 off	165,412	4	4	Y		5,954,832
Data & Comms Technicians	9 on/5 off	135,901	2	2	Y		2,718,020
Mining Surveyor Snr	8 on/6 off	214,975	1	1	Y	Y	2,149,750
Mining Surveyor	8 on/6 off	172,262	1	1	Y		1,722,620
Survey Technician	8 on/6 off	114,473	2	2			2,289,460
Senior Geotechnical Engineer	8 on/6 off	214,975	1	1	Y	Y	2,149,750
Managers Admin. Asst.	5 on/2 off	104,253	1	1	Y		1,042,530
Senior Geology Technicians	8 on/6 off	148,094	2	2	Y	Y	2,665,692
Geology Technicians	5 on/2 off	118,476	4	6			5,686,848
<b>Subtotals</b>	<b>Personnel</b>		<b>24</b>	<b>26</b>	<b>24</b>	<b>10</b>	
	<b>Annual Cost (\$)</b>		<b>4,047,200</b>	<b>4,284,152</b>			<b>40,461,972</b>

Unit costs for general administration are provided in Table 10-7. The rates are shown as per unit or person per period where possible and as an allowance per month otherwise.

Table 10-7 General Overheads and Consumables

Item	Unit	Rate (\$)	Total (\$)
Drone & Survey Equipment	person / month	1,800	435,600
Consumables (all mining activities)	person / month	80	234,400
Training, Courses & Conferences	month	8,333	1,008,293
Slope Monitoring	month	8,333	1,008,293
Consultants - General	month	8,333	1,008,293
General Mine Planning Software	month	4,773	577,535
<b>TOTAL</b>			<b>4,272,414</b>

## 10.4.2 Contractors Overheads

The mining personnel employed by Contractor will be required to oversee and manage the day to day activities of the mining operation and equipment maintenance.

Table 10-8 Contractor Monthly Management Fees & Overheads

Component	Rate (\$/month)	TOTAL (\$)
Management, Supervision & Administration - Operations	439,010	45,218,060
Management, Supervision & Administration - Maintenance	220,597	22,721,514
Light Vehicles	103,548	10,665,450
Workshop Support Equipment	86,867	8,947,332
Contractor Facilities Recovery	45,067	4,686,079
Maintain & Repair Contractor's Facilities	22,860	2,354,561
Explosives Storage & Downhole Service	210,236	21,654,278
Computing & Communication	39,910	4,110,780
Pit Dewatering	17,296	1,781,450
<b>Total</b>	<b>1,185,391</b>	<b>122,139,504</b>

## 10.4.3 Mine Development

The mine development costs include, clearing, topsoil removal and road construction. The unit rate for these activities is shown in Table 10-9.

Table 10-9 Mine Redevelopment Unit Rates

Activity	Unit	Rate	Quantity	Cost (\$)
Clear and Grub Vegetated Areas	\$/Ha	2,717.34	653	1,829,382
One Way Haul Distance < 500m	\$/bcm	5.62	493,609	3,043,396
One Way Haul Distance > 1,000m	\$/bcm	7.03	1,464,178	10,391,125
Heavy Haulage Road Construction (mine waste subgrade, select materials base course 300mm thick)	\$/km	181,032	9.0	1,900,838
<b>TOTAL</b>				<b>17,164,741</b>

## 10.4.4 Grade Control

Grade control drilling rates were based on recent quotations for a project of similar size at \$35/lin.m based 28 days per month on 10 hour shift with fuel and accommodation for personnel provided by Ausgold. The estimated penetration rate was 250 lin.m/shift with a fuel consumption rate of 450 L/shift.

Assay costs were provided by Ausgold at \$12.50/sample for fire assay.

The drilling and assay quantities and costs are shown in Table 10-10 and Table 10-11 respectively.

Table 10-10 Grade Control Drilling quantities (including Pre-production)

Bench	Dingo1 (lin.m)	Dingo2 (lin.m)	Jack1 (lin.m)	Jack2 (lin.m)	Jack3 (lin.m)	Jack4 (lin.m)	Jinkas1a (lin.m)	Jinkas1b (lin.m)	Jinkas2 (lin.m)	Jinkas3 (lin.m)	Jinkas4 (lin.m)	Jinkas5 (lin.m)	Jinkas6 (lin.m)	Olympia1 (lin.m)	Olympia2 (lin.m)	Olympia3 (lin.m)	Olympia4 (lin.m)	Olympia5 (lin.m)	Total Drill metres	GC Drill Opex Cost (\$)
389											6								6	194
369							1				24,341	149	4						24,494	857,297
349	260			145			330	586		6,229	53,534	3,923	17,037	42	392	833	236	142	83,689	2,929,130
329	45,419	7,087	6,090	8,148	4,705	3,781	11,522	5,728	629	10,716	22,521	15,340	19,641	2,421	719	2,074	2,939	1,001	170,481	5,966,826
309	46,632	24,416	14,848	10,776	11,175	8,498	16,419	12,939	1,310	12,433	6,769	26,082	8,515	278		196	658		201,944	7,068,036
289	27,563	10,146	12,322	5,089	3,893	5,848	4,649	16,532	7,662	11,676	4,648	33,264	3,150						146,442	5,125,453
269	17,816	2,862	6,633	1,576	31	3,050		11,278	14,367	11,042		30,245	177						99,077	3,467,708
249	10,382	258	1,668	349		1,434		1,173	19,439	7,863		18,419							60,984	2,134,450
229	3,701		171			251			11,754	6,723		11,277							33,877	1,185,706
209	997									9,220		1,615							11,831	414,100
189										5,790									5,790	202,654
169										6,207									6,207	217,231
149										2,304									2,304	80,652
<b>Total</b>	<b>152,770</b>	<b>44,768</b>	<b>41,731</b>	<b>26,083</b>	<b>19,804</b>	<b>22,863</b>	<b>32,920</b>	<b>48,237</b>	<b>55,161</b>	<b>90,203</b>	<b>111,818</b>	<b>140,313</b>	<b>48,523</b>	<b>2,741</b>	<b>1,111</b>	<b>3,103</b>	<b>3,834</b>	<b>1,144</b>	<b>847,127</b>	<b>29,649,437</b>

Table 10-11 Grade Control Assay quantities (including Pre-production)

Bench	Dingo1 #	Dingo2 #	Jack1 #	Jack2 #	Jack3 #	Jack4 #	Jinkas1a #	Jinkas1b #	Jinkas2 #	Jinkas3 #	Jinkas4 #	Jinkas5 #	Jinkas6 #	Olympia1 #	Olympia2 #	Olympia3 #	Olympia4 #	Olympia5 #	Total Samples	GC Assay Opex cost (\$)
389										1									1	18
369							0			6,526	31	1							6,558	81,981
349	0			28		63	68		1,928	20,634	918	4,714	14	87	160	56	21		28,690	358,624
329	1,205	58	2,202	3,186	1,040	6,214	1,115	83	3,729	8,384	4,373	6,859	952	380	863	1,368	274	1,453	43,737	546,718
309	7,389	3,042	6,019	5,656	2,405	9,170	3,461	222	4,242	2,482	10,748	3,085	82		99	346		3,845	62,293	778,657
289	3,628	2,795	4,901	2,966	1,531	2,907	7,978	2,426	4,688	3,248	16,947	1,371						1,745	57,131	714,133
269	3,023	1,003	2,877	986	674		8,447	5,444	6,011		20,881	45						8	49,398	617,481
249	4,159	94	958	195	801		905	9,482	5,292		15,028								36,913	461,417
229	2,442		80		117			4,997	4,602		10,936								23,173	289,661
209	947								4,100		720								5,767	72,091
189									3,111										3,111	38,885
169									3,617										3,617	45,211
149									2,719										2,719	33,993
<b>Total</b>	<b>22,794</b>	<b>6,991</b>	<b>17,037</b>	<b>13,017</b>	<b>6,567</b>	<b>18,355</b>	<b>21,974</b>	<b>22,654</b>	<b>44,038</b>	<b>41,275</b>	<b>80,581</b>	<b>16,075</b>	<b>1,048</b>	<b>466</b>	<b>1,121</b>	<b>1,770</b>	<b>295</b>	<b>7,051</b>	<b>323,110</b>	<b>4,038,869</b>

## 10.4.5 Drill and Blast

Contract rates for drill and blasts are presented in Table 10-12.

Table 10-12 Drill & Blast Unit Rates

Description	Drilling (\$/lin.m)	Drilling (lin/m)	Drill cost (\$)	Blasting (\$/bcm)	Blast Volumes (bcm)	Blast cost (\$)
Ore Oxide	14.41	56,142	813,961	0.69	2,412,299	1,671,925
Ore Transitional	16.74	51,546	942,998	0.79	1,812,152	1,560,275
Ore Fresh	24.99	420,933	10,554,693	1.55	8,139,010	12,642,574
Waste Oxide	14.55	615,897	8,986,341	0.68	24,913,196	17,002,837
Waste Transitional	16.95	565,971	9,916,999	0.85	20,604,295	18,113,038
Waste Fresh	25.45	2,576,118	65,747,887	1.52	46,892,070	71,519,207
<b>TOTALS</b>		<b>4,286,607</b>	<b>96,962,879</b>		<b>104,773,022</b>	<b>122,509,856</b>
Presplits				\$526.40/hole	40,900 holes	21,529,728

## 10.4.6 Load and haul

Total Load and haul costs for the operations were:

- Ore to ROM - \$75,199,299 at an average cost of \$7.95/bcm
- Ore to Stockpile - \$22,226,056 at an average cost of \$7.65/bcm
- Waste to WRD - \$608,085,087 at an average cost of \$6.58/bcm

Contract rates for load and haul were provided for all 17 pit/stage in 10m bench increments. The unit rates provided by Contractor 2 are presented in:

- Table 10-13 for ore hauled to the ROM pad
- Table 10-14 for ore hauled to stockpile
- Table 10-15 for waste hauled to waste dumps.

Table 10-13 Ore haulage unit rate to ROM pad (\$/bcm)

Bench Toe RL	Jinkas1a	Jinkas1b	Jinkas4	Jinkas5	Jackson2	Jackson4	Dingo1	Dingo2	Olympia1
389	7.86		3.85	4.79					
379	7.00	7.14	3.85	4.61					
369	5.46	5.51	3.68	4.79					
359	5.01	5.64	3.84	4.79	7.19				
349	8.58	4.99	5.00	5.99	5.81	9.70	4.97		5.85
339	8.98	6.35	5.21	7.29	5.87	7.94	4.97	5.68	6.31
329	7.71	7.18	6.20	7.77	6.07	7.91	5.00	5.47	8.03
319	8.05	7.59	6.49	8.38	7.19	8.86	5.20	5.51	8.58
309	8.38	8.05	6.83	8.38	8.77	8.46	5.76	5.74	
299	8.38	8.07	7.03	8.51	9.04	10.59	7.13	5.89	
289		9.07	7.03	8.79	9.78	10.27	7.85	6.12	
279		9.06		9.06	9.62	9.99	8.35	8.57	
269		9.13		9.06	10.81	9.88	8.98	9.17	
259		9.40		9.31	11.06	9.88	9.34	9.44	
249				9.83		9.72	9.34		
239				10.06		9.17	10.25		
229				10.20			10.55		
219				10.47			10.92		
Bench Toe RL	Jinkas2	Jinkas3	Jinkas6	Jackson1	Jackson3	Olympia2	Olympia3	Olympia4	Olympia5
384	6.73	5.04							
374	6.73	4.79	5.72						
364	6.94	4.79	4.62				5.88		
354	8.23	5.36	4.94			5.82	5.64	5.77	6.45
344	8.96	6.74	5.12	6.15	6.53	6.00	5.65	5.77	5.41
334	8.67	6.94	5.17	6.03	6.01	6.11	5.83	7.43	7.33
324	8.87	7.73	6.88	6.40	6.26		6.31	7.92	7.87
314	9.46	8.38	7.24	7.40	8.06		8.24	8.52	
304	8.13	9.11	8.10	9.16	9.26				
294	9.15	8.68	8.11	9.29	9.46				
284	9.46	9.06	7.97	8.89	9.46				
274	9.46	9.06		9.52					
264	9.50	7.34		9.76					
254	9.92	9.42		10.51					
244	11.16	9.71		10.76					
234	11.57	9.90		11.01					
224	11.70	10.11							
214		10.38							
204		9.67							
194		9.85							
184		10.06							
174		10.33							

Table 10-14 Ore haulage unit rate to Stockpile (\$/bcm)

Bench Toe RL	Jinkas1a	Jinkas1b	Jinkas4	Jinkas5	Jackson2	Jackson4	Dingo1	Dingo2	Olympia1
389	389	5.67		3.53	5.04				
379	379	6.35	5.35	3.54	4.96				
369	369	4.87	5.30	3.33	4.96				
359	359	4.54	5.25	3.45	6.15	7.33			
349	349	6.28	4.78	4.26	6.15	5.97	7.33	5.97	
339	339	8.47	5.93	5.14	6.82	7.30	6.81	5.97	3.44
329	329	7.23	6.74	5.98	8.17	7.11	7.76	5.60	6.31
319	319	7.34	6.98	5.97	7.34	7.58	8.28	7.26	6.31
309	309	7.40	7.39	5.92	7.34	7.62	8.28	7.83	6.31
299	299	7.67	7.74	6.35	7.59	8.15	8.66	8.17	6.31
289	289		8.38	6.37	8.38	8.68	8.75	8.90	6.52
279	279		8.05		8.38	9.08	8.75	6.87	8.42
269	269		8.61		8.47	9.08	8.75	9.00	9.08
259	259		8.29		8.93	9.08	8.75	10.00	9.00
249	249				9.34		8.75	10.11	
239	239				9.34		8.66	9.60	
229	229				9.51			9.08	
219	219				9.79			9.08	
Bench Toe RL	Jinkas2	Jinkas3	Jinkas6	Jackson1	Jackson3	Olympia2	Olympia3	Olympia4	Olympia5
384	7.09	5.43							
374	7.09	5.12	4.13						
364	7.55	5.43	5.12				6.34		
354	8.64	5.76	7.09			5.47	5.21	5.21	6.73
344	8.67	6.25	6.31	6.80	6.80	5.64	5.30	5.42	6.64
334	8.57	6.52	6.80	6.96	5.64	5.64	6.31	6.03	6.74
324	8.76	7.23	8.05	7.92	6.58		7.55	7.75	8.75
314	8.75	7.70	8.66	7.92	7.92		8.20	8.16	
304	7.74	8.75	9.08	8.56	8.22				
294	8.75	5.94	9.08	8.75	8.78				
284	8.75	8.05	9.00	8.86	9.07				
274	8.50	8.05		9.26					
264	8.36	8.64		9.47					
254	8.43	9.06		9.72					
244	8.75	9.06		9.62					
234	8.75	9.12		9.86					
224	8.87	9.38							
214		8.89							
204		9.46							
194		9.46							
184		9.46							
174		9.65							

Table 10-15 Waste haulage unit rate to WRD (\$/bcm)

Bench Toe RL	Jinkas1a	Jinkas1b	Jinkas4	Jinkas5	Jackson2	Jackson4	Dingo1	Dingo2	Olympia1
389	8.18		5.07	5.56					
379	7.47	7.76	5.59	5.38					
369	5.87	6.01	4.16	5.53					
359	5.60	6.14	4.32	5.93	5.52				
349	6.72	6.83	4.70	7.28	4.44	5.60	3.64		3.72
339	8.45	7.28	6.03	8.08	4.69	5.26	3.64	4.29	3.80
329	8.06	6.03	6.47	8.84	4.86	5.07	3.92	4.52	5.27
319	6.30	6.36	6.72	8.59	5.88	6.05	4.36	4.53	5.45
309	6.82	6.64	7.09	8.74	6.49	8.07	5.36	4.53	
299	7.09	7.09	7.20	8.93	6.50	7.20	6.36	4.75	
289		7.61	7.60	9.50	6.74	7.20	5.89	5.14	
279		7.29		9.75	7.43	7.52	6.13	6.95	
269		7.56		9.99	8.09	7.80	6.67	7.46	
259		7.71		9.24	8.13	7.80	6.73	7.71	
249				10.75		7.87	6.73		
239				10.37		8.18	7.80		
229				10.56			6.84		
219				10.79			8.10		
Bench Toe RL	Jinkas2	Jinkas3	Jinkas6	Jackson1	Jackson3	Olympia2	Olympia3	Olympia4	Olympia5
384	6.35	4.18							
374	6.23	3.91	6.77						
364	6.39	4.25	5.38				3.73		
354	7.39	4.65	5.50	3.73		3.17	3.46	3.73	4.79
344	7.93	6.03	5.61	4.01	4.64	3.07	3.46	3.78	3.95
334	8.04	6.03	6.14	4.73	4.86	3.36	3.64	4.18	5.11
324	8.04	6.40	7.61	5.45	4.97		4.12	4.49	5.64
314	8.04	6.68	8.40	5.75	5.97		5.45	5.45	6.17
304	7.15	8.09	8.97	6.60	6.74				
294	7.70	6.84	9.33	8.03	6.84				
284	8.04	7.07	9.33	6.24	7.11				
274	8.04	7.44		6.24					
264	8.11	7.44		6.38					
254	7.94	7.48		6.66					
244	8.61	8.01		6.84					
234	8.77	8.74							
224	8.97	8.04							
214		8.14							
204		8.68							
194		8.83							
184		9.07							
174		9.65							

## 10.4.7 Ore Rehandle

Ore rehandle rates were requested for ore feed to the crusher and reclaim from stockpile. Rates provided are presented in Table 10-16.

**Table 10-16 Rehandle Unit Rates**

	bcm	\$/bcm	Cost (\$)
Reclaim (FEL within 100m)	9,456,161	2.41	22,811,336.13
Rehandle (Truck & Loader 1.6km)	3,293,030	6.39	21,403,631.89
	<b>12,749,191</b>		<b>44,214,968</b>

## 10.4.8 Rehabilitation

The unit rates for rehabilitation provided by Contractor 2 are presented in Table 10-17.

**Table 10-17 Rehabilitation Unit Rates**

Activity	Unit	Rate	Quantity	Cost (\$)
Profile Outer Surface of Waste Dump (18 degree slope angle & 8 m wide berms)	\$/Ha	2,112.87	232	490,186
Profile Top Surface of Waste Dump	\$/Ha	1,408.58	104	146,493
Contour ripping of outer surface of waste dump	\$/Ha	1,690.30	232	392,149
One Way Haul Distance of 1,000 m	\$/bcm	2.11	504,000	1,062,714
Spread and Trim Topsoil to Line & Level	\$/Ha	2,113	336	709,925
<b>TOTAL</b>				<b>2,801,467</b>

## 10.4.9 Fuel Consumption

The contractors were requested to provide an estimate for fuel consumption. Grade control fuel was derived from estimated penetration rates and fuel burn for the RC drill rig. Quantities are presented in Table 10-18.

**Table 10-18 Fuel Consumption**

Cost Centre	Litres	Cost (\$)
Excavator Fleet	24,749,116	27,486,499
FEL Fleet	5,535,769	6,089,346
Dump Truck Fleet	102,659,839	114,187,493
Mining Support Equipment	19,370,670	21,534,992
Dewatering Equipment	2,916,611	3,208,272
Other Support Equipment	5,868,885	6,490,861
Bulk Explosives	21,184,801	23,395,950
GC Drilling	1,524,828	1,707,128
<b>TOTAL</b>	<b>183,810,519</b>	<b>204,100,541</b>

## 10.5 Mining Cost Summary

Total capital and operating costs were estimated at \$1,471 M or \$5.30/t mined.

Capital cost totalled \$43.3 M comprised of:

- Owner Equipment - \$0.61 M
- Contractor Site Establishment - \$2.90 M
- Contractor Mobilisation - \$2.87 M
- Pre-production mining cost - \$36.89 M

Total operating cost for the life of mine were estimated at \$1,428 M as summarised in Table 10-19.

**Table 10-19 Annual Mining Operating Cost Summary by Cost Centre**

OPEX (\$M)	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total (\$M)
Clear & Grub	0.1	0.4	0.3	0.3	0.4	0.1	0.1	0.1	0.0	0.09		1.8
Topsoil removal	0.4	2.8	2.1	2.0	3.1	0.4	0.8	0.9	0.4	0.6		13.4
Mine Haul Roads	0.3			0.7	0.1	0.1		0.7				1.9
Owner Overheads	0.4	4.7	4.7	4.7	4.7	4.7	4.7	4.5	4.5	3.8	3.8	45.1
Contractor Overheads	1.2	14.0	14.0	14.0	14.0	14.0	14.0	10.5	10.5	7.0	7.0	120.4
L&H Waste	7.6	81.8	76.4	76.5	76.4	86.5	80.3	42.5	28.5	28.5	30.6	615.7
L&H Ore	1.8	9.9	9.5	13.6	12.9	7.1	12.8	14.3	8.0	4.7	4.6	99.2
D&B Waste	1.4	27.3	29.2	30.5	25.7	27.9	29.0	15.4	10.2	8.5	7.7	212.8
D&B Ore	0.3	3.2	2.8	3.7	3.5	2.4	4.2	4.0	2.0	0.9	1.2	28.2
Fuel Consumption	1.9	26.6	25.3	26.1	24.2	26.2	30.2	15.3	12.9	8.3	7.1	204.1
Ore Rehandle	0.4	5.6	4.4	3.8	4.0	5.3	3.8	3.1	4.7	6.6	2.5	44.2
Grade Control	0.6	3.1	2.8	3.7	5.0	2.8	4.6	4.7	2.3	2.7	2.0	34.3
Pit Dewatering	0.0	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	1.8
Mine Rehabilitation						0.5				0.3	2.0	2.8
Demobilisation			0.03		0.03		0.02	0.3	0.1	0.2	1.5	2.2
<b>TOTAL (\$M)</b>	<b>16</b>	<b>180</b>	<b>172</b>	<b>180</b>	<b>174</b>	<b>178</b>	<b>185</b>	<b>117</b>	<b>84</b>	<b>72</b>	<b>70</b>	<b>1,428</b>

# 11 FINANCIALS

Table 1-10 summarises the EBITDA financials related to the open pit by year. Total operating costs inclusive of mining, processing, and selling costs are \$2,491 M and together with the total capital costs of \$368 M, the total cost base is \$2,875 M. Total revenue is \$3,412 M based on 1,137 koz of gold recovered using the base gold price of \$3,000/oz. Hence, a final cashflow before taxes and depreciation of \$537 M can be generated with a project IRR of 33% and a payback within just over 4 years.

All In Sustaining Costs (ASIC) are \$2,204 per recovered ounce for the project. Figure 11-1 illustrates the cost per ounce over the life of the project and Figure 11-2 outlines the cashflow and cumulative discounted cashflow for the project.

Figure 11-1: ASIC per ounce

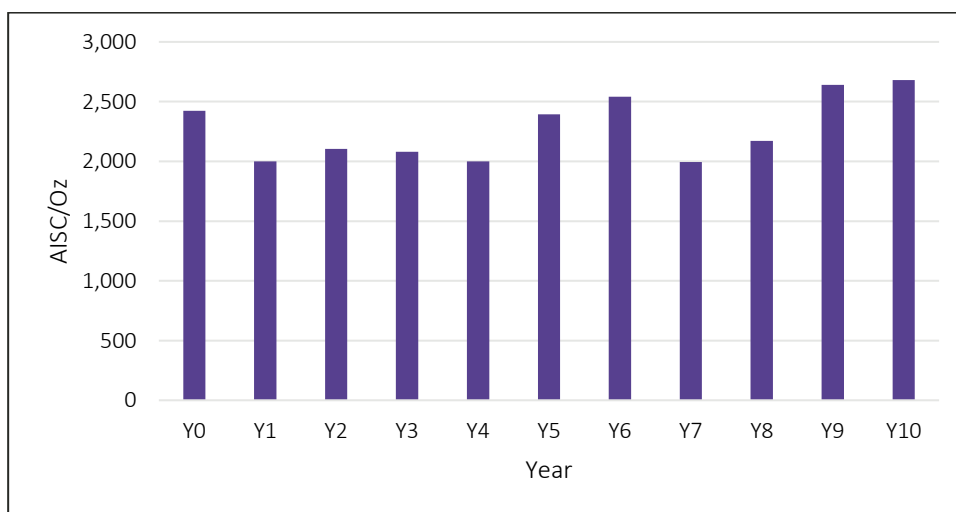


Figure 11-2: Cumulative Cashflow

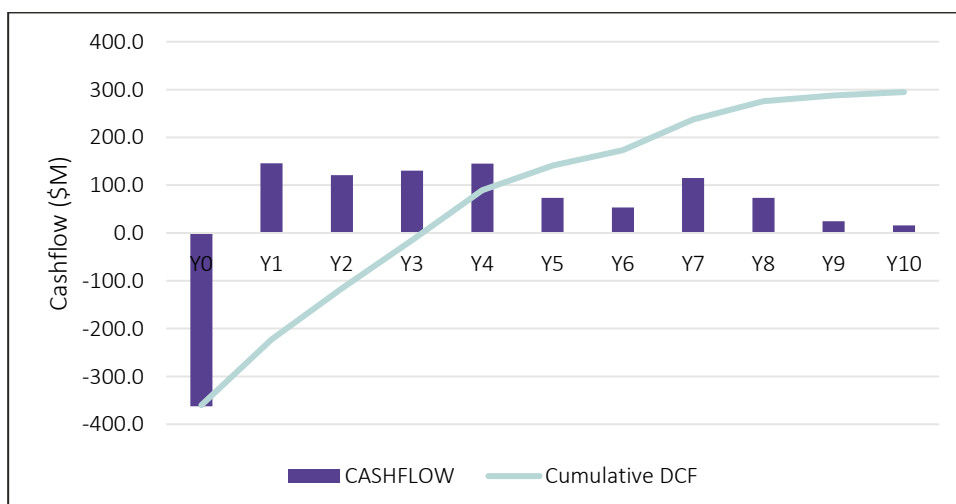


Table 11-1: EBIT Level Cashflow

Metric	Unit	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
Rock Mined	Mt	9.798	34.547	34.652	35.441	35.112	35.112	31.022	21.579	15.296	12.755	12.165	<b>277,480</b>
Gold produced	Koz	0.009	0.146	0.136	0.142	0.145	0.122	0.116	0.114	0.089	0.069	0.049	<b>1,137</b>
Revenue	\$M	26.6	438.8	406.5	424.9	435.8	365.1	349.3	342.7	267.5	206.5	148.1	<b>3,411.6</b>
<b>Operating Cost</b>													
Mining Cost	\$M	16.3	179.5	171.9	179.7	174.4	178.2	184.7	116.6	84.3	72.2	70.1	<b>1,427.9</b>
Process Cost	\$M	4.5	100.8	102.1	103.2	104.0	103.1	101.4	101.9	102.0	103.8	58.1	<b>985.1</b>
Selling Cost	\$M	0.7	12.0	11.2	11.7	12.0	10.0	9.6	9.4	7.3	5.7	4.1	<b>93.6</b>
Total Opex	\$M	21.5	292.4	285.2	294.5	290.4	291.3	295.7	228.0	193.6	181.7	132.3	<b>2,506.6</b>
<b>Capital Cost</b>													
Mining	\$M	42.8	0.3	0.2				0.01					<b>43.3</b>
Process	\$M	325.0											<b>325.0</b>
Total Capex	\$M	367.8	0.3	0.2				0.01					<b>368.3</b>
CASHFLOW	\$M	-362.7	146.1	121.1	130.3	145.4	73.8	53.6	114.8	73.8	24.8	15.8	<b>536.7</b>
Discounted Cashflow (8%)	%	-360.1	137.6	106.4	101.3	104.6	51.6	32.0	64.3	38.3	11.9	7.0	<b>294.8</b>
Cumulative DCF	\$M	-360.1	-222.5	-116.1	-14.8	89.8	141.4	173.4	237.6	275.9	287.8	294.8	
Average cost per Ounce	\$/oz	2,424	1,999	2,105	2,080	1,999	2,394	2,540	1,996	2,172	2,640	2,680	<b>2,204</b>

## 12 ORE RESERVE

### 12.1 Site Visit

A site visit was completed by a Hossein Parhoon, a Senior Consultant for the Orelogy Consulting team, on the 16-17<sup>th</sup> October 2024.

The site visit toured the extents of the mining tenement, the existing Jinkas Hill and Dingo Open pits and historical infrastructure. The visit concluded with a tour of the core farm where the barren light coloured waste was observed to clearly contrast with the dark coloured ore as illustrated in Figure 12-1.

Figure 12-1 Remaining Core after Testing



### 12.2 Competent Person

Orelogy Consulting Pty Ltd was responsible for the mining component of the Katanning Gold Project Feasibility Study. As a result, Orelogy have developed an open pit Ore Reserve Estimate as of 31st March 2025. The Ore Reserve was prepared in accordance with the guidelines of the Joint Ore Reserve Committee (JORC 2012). Jake Fitzsimons (MAusIMM), Principal Consultant, is the Competent Person for the Ore Reserve.

## 12.3 Ore Reserve

Mineral Resources were converted to Ore Reserves in line with the material classifications which reflect the level of confidence within the resource estimate. The Ore Reserve reflects that portion of the Mineral Resource which can be economically extracted by open pit methods. The Ore Reserve considers the modifying factors and other parameters outlined in the preceding sections of this report and summarised in the following paragraphs, including but not limited to the mining, metallurgical, social, environmental, statutory and financial aspects of the project.

The Ore Reserve includes an allowance for mining dilution and ore loss. Orelogy developed open pit mining models with an average dilution of 26% and an average ore loss of 25% for the CZ and SZ pits.

In line with the JORC 2012 guidelines, the Proved Ore Reserve estimate is based on mineral resources classified as Measured and the Probable Ore Reserve is based on resources classified as Indicated.

The reported Mineral Resource estimate is inclusive of the resources converted to Ore Reserves. The Katanning Gold Project Feasibility Study Ore Reserve as outlined in Table 12-1 was reported using variable cut-off grades as summarised in Table 5-4. The cut-off grades vary by material type due to variations in process recoveries and costs.

Table 12-1: Ore Reserve

	Category	Ore (Mt)	Au Grade (g/t)	Contained Gold (koz)
Central Zone	Proved	27.6	1.15	1,015.3
	Probable	4.8	0.96	146.9
	Sub-total	32.3	1.12	1,162.2
South Zone	Proved	1.2	0.97	36.5
	Probable	1.7	1.01	54.6
	Sub-total	2.9	0.99	91.0
<b>TOTAL</b>		35.2	1.11	1,253.2

## 13 RISKS AND OPPORTUNITIES

### 13.1 Execution Risks

The following risks need to be considered as part of the next phase of work and include:

- Geotechnical stability of the overall slope angles is based on 90° batters which were adjusted to 80° batters with narrower berms to facilitate mine design. It is recommended that the slope design criteria is reviewed and designs updated for implementation.
- Develop a detailed NAG/PAG (i.e. non-acid versus potentially acid producing) waste dumping strategy. This may become an issue as more deeper sulphides are mined.
- The Warren Road crossing requires detailed design and development of a traffic management plan to minimise the risk to the public.
- The mine is surrounded by farmland which will require management of dust and sound particularly at night.

### 13.2 Mining Cost Risks

As with all mining projects, there is a risk that mining costs will be more than anticipated in this cost estimate due to the following:

- The rock is strong and highly competent which may not fracture as well as modelling has shown using modified Kuzram methods. Improved fragmentation may require tighter drilling patterns or higher density explosives.
- Inflationary influences causing the cost of fuel, parts and labour to increase.

### 13.3 Execution Opportunities

There are numerous opportunities identified with the development of the project that will be evaluated and integrated with existing or continued technical work including:

- Feedback from one of the Mining Contractors recommended changing the waste drilling hole diameter from 172mm to 165mm as this increases the options for drill selection and provides consistence in blast design.
- Two of the contractors recommended an increase in truck size from 140 t to 180 t. The potential cost savings from the larger trucks is likely to provide a small mining cost reduction.

- Due to space limitations, the Main waste dump is located to the south east of the Jinkas pits. Expanding the tenements to the East of the Jinkas/White Dam will reduce the haulage distance for waste.
- Backfilling of the Olympia pits has been exploited to minimise ex-pit hauls and mining costs from Jackson, however, there is potential for a larger pit at Olympia at prices above \$4000/oz. Expansion of the tenements to provide additional areas for waste storage is recommended to avoid potential sterilisation of this ore.

## 14 BIBLIOGRAPHY

- DISR. (2011, July). *A guide to leading practice in sustainable development in mining*. Retrieved from Australian Government Department of Industry, Science and Resources: <https://www.industry.gov.au/sites/default/files/2019-04/lpsdp-a-guide-to-leading-practice-sustainable-development-in-mining-handbook-english.pdf>
- Kharitonov, A. (2025, 3 5). *tradersunion.com*. Retrieved from [tradersunion.com/currencies/forecast/aud-usd/long-term-forecast/](https://tradersunion.com/currencies/forecast/aud-usd/long-term-forecast/): <https://tradersunion.com/currencies/forecast/aud-usd/long-term-forecast/#:~:text=According%20to%20analytical%20forecasts%2C%20the%20price%20of%20AUD%2FUSD,to%20be%20%240.5433%20by%20the%20end%20of%202029.>
- Sepanek, E. (2025, 3 5). *Macquarie Group Lifts 2025 Gold Forecast to \$3,000*. Retrieved from Scottsdale Bullion & Gold: <https://www.sbcgold.com/blog/macquarie-group-lifts-2025-gold-forecast-to-3000/#:~:text=Gold%20has%20the%20momentum%20to%20clear%20more%20all-time,the%20potential%20for%20reaching%20%243%2C000%20by%20year%20E2%80%99s%20end.>